Kansanshi Operations

North West Province, Zambia
NI 43-101 Technical Report

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David Gray (QP) *BSc(Geology), MAusIMM, PrSciNat(SACNASP),* Group Mine and Resource Geologist, FQM (Australia) Pty Ltd

Michael Lawlor (QP) *BEng Hons (Mining), MEngSc, FAusIMM*, Consultant Mining Engineer, FQM (Australia) Pty Ltd

Robert Stone (QP) *BSc(Hons), CEng, ACSM*, Technical Manager, FQM (Australia) Pty Ltd

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ITEM 1 SUMMARY

This Technical Report on the Kansanshi Operations (the property) has been prepared by Qualified Persons David Gray, Michael Lawlor and Robert Stone of First Quantum Minerals Pty Ltd (FQM, the issuer) for FQM.

This report supersedes the NI 43-101 Technical Report dated December 2012 (the 2012 Technical Report) which was filed as two separate documents under an FQM covering letter. The December 2012 Mineral Resource related aspects were addressed by Malcolm Titley (Qualified Person, QP) of CSA Global (UK) Ltd (Titley, CSA, 2012), whilst the Mineral Reserve related aspects were addressed by Nick Journet and Tony Cameron (joint QPs) of DumpSolver Pty Ltd (Journet and Cameron, DumpSolver, 2012).

In this Technical Report, updated for new information as at 31st May 2015, the respective Mineral Resource and Mineral Reserve aspects were addressed by FQM employees David Gray (QP) and Michael Lawlor (QP). Metallurgical and mineral processing aspects were addressed by FQM employee Robert Stone (QP). Malcolm Titley of CSA was a contributing author for items related to the Mineral Resource estimates.

1.1 Property location and ownership

The Kansanshi Operations are located approximately 15 km north of the town of Solwezi, the capital of the Zambian North West Province. The site is 18 km south of the Democratic Republic of Congo (DRC) border. Chingola, a major town in the Zambian Copperbelt, is approximately 180 km to the southeast of Solwezi.

First Quantum Minerals Ltd (FQM or the Company) has an 80% interest in the Kansanshi Project, through a subsidiary operating entity, Kansanshi Mining PLC (KMP). The remaining 20% is owned by a subsidiary of Zambian Consolidated Copper Mines (ZCCM).

1.2 Project background

Mining at Kansanshi is carried out in two open pits, the Main and North West Pits, using conventional open pit methods, with electric and hydraulic excavators and a mixed fleet of haul trucks. A near-by deposit has been defined at South East Dome, with commencement of mining scheduled for 2017.

Ore processing is flexible to allow for variation in plant feed either through a solvent extraction electrowinning (SXEW) oxide leach circuit, a sulphide flotation circuit, or a transitional circuit (for mixed ore feed) with facilities to beneficiate flotation concentrate to final cathode via a High Pressure Leach (HPL) circuit. The flexibility of the Kansanshi processing facilities allows any ore type to be treated through any of the three circuits.

Expansion of the leach circuit in 2014 effectively doubled the leach capacity and matched the output of low cost sulphuric acid from a now completed first stage on-site smelter. This allows ore previously classified as mixed to be leached, reducing the plant feed types to essentially two, ie float/leach and sulphide. The current treatment capacity at Kansanshi is 15.6 Mtpa of float/leach feed and 13.2 Mtpa of sulphide feed.

The first smelting furnace was commissioned in late 2014, with a capacity of 1.2 million tonnes of concentrate. The two direct benefits of this smelter are the flexibility to overcome concentrate stockpiling in response to continued third party capacity constraints and to provide sulphuric acid as a low cost by-product. The ready supply of acid is an important benefit to SXEW processing of the oxide ores, especially high-acid consuming ore types and some ores which would otherwise be processed as mixed feed. The proposed construction of a second smelting furnace was placed on hold in late 2014 pending changes in the Zambian tax and royalty regimes (refer to Item 1.7).

Going forward, the Project will produce cathode plus anode product, in place of cathode plus concentrate product.

Further capital works also commenced in early 2010 to further increase the sulphide flotation capacity to around 25 Mtpa (the S3 expansion project). The S3 project includes the construction of an overland conveyor route from a near-mine surface transfer bin receiving crushed ore from proposed semi-mobile, in-pit crushers. Progress on S3 was deliberately placed on hold in late 2014, however, for the same Zambian tax and royalty reasons as mentioned above.

This Technical Report now addresses the future advancement of the S3 project, with commissioning in mid 2017 incorporated into the life of mine production schedule.

1.3 Project approvals

All surface rights necessary to develop and operate the Project have been obtained and include four leases covering in excess of 7,000 ha. The right to mine is governed by a large-scale mining licence (LML 16) granted in March 1997, which has a term of 25 years and can be renewed upon application. In December 2000 the licence area was increased from 4,244 ha to 21,593 ha, and in November 2014 was further increased to 24,865 ha.

1.4 Geology and mineralisation

The Kansanshi deposit, currently mined from the Main Pit and the North West Pit, is hosted by deformed metasediments of the Lower Kundulungu Group, within the Katanga Supergroup of the Zambian Central African Copperbelt. The individual rock units of the stratigraphic sequence comprise dolomites, dolomitic marbles, and various schists and phyllites. The recently defined South East Dome and Rocky Hill deposits, in the south east of the tenement area, are similarly hosted but separated from the Main Pit by a structural discontinuity.

Copper mineralisation at Kansanshi occurs within two domal structures along the crest of a regional antiform. These domes are closely associated with the deposit mineralisation been mined in the Main and North West Pits. Three styles of primary sulphide mineralisation are associated with these domes:

- stratified and disseminated mineralisation
- vertically dipping, quartz-carbonate-sulphide veins crosscutting the stratigraphy
- breccia mineralisation

Oxidation of primary mineralisation is associated with depth of weathering, and is reflected as:

- nearer surface weathering around vertical veins, with oxide mineralisation evident as malachite and tenorite

- mixed mineralisation, with a wide variety of copper ore mineralogy, occurs in a wide transitional zone between the base of complete oxidation and the top of fresh rock

Mineralisation at South East Dome and Rocky Hill occur as higher grades associated with near vertical brecciation and veins as well as lower grades associated with stratigraphic controlled mineralisation.

Primary copper sulphide mineralisation is dominated by chalcopyrite, with very minor bornite and is accompanied by relatively minor pyrite and pyrrhotite. Oxide mineralisation is dominated by chrysocolla with malachite, limonite and cupriferous goethite. The mixed zone includes both oxide and primary mineralisation but also carries significant chalcocite, minor native copper and tenorite. Some copper appears to be hosted in clay and mica minerals, and is essentially refractory. Gold mineralisation appears to occur in association with copper.

1.5 Metallurgical summary

The Kansanshi process plant has been designed to operate with a high degree of flexibility to suit the various ore types delivered from the mined orebodies. The three main process routes are for treating sulphide, mixed and oxide ores independently. The flexibility of the circuitry, however, allows any ore type to be treated through any of the three circuits. This allows balancing of the tonnages as each circuit has a different inherent capacity.

The SXEW (leach) circuit was subject to an upgrade in 2014 to effectively double the leach capacity. This expansion will match the output of low cost acid from the Kansanshi smelter and allow ore previously classified as mixed to be leached, thereby reducing the plant feed types to essentially two, ie float/leach and sulphide.

The S3 expansion to the sulphide circuit whilst currently on hold, is a stand-alone facility and does not impact upon the operation of the remainder of the Kansanshi process plant.

1.6 Exploration status

Since the 2012 Technical Report, FQML has commissioned a number of geophysical surveys to explore for additional copper mineralisation as well as to help define the limits of the Kansanshi mineralisation system. These have included:

- Three north east oriented 2D seismic lines completed north of North West pit that yielded poor results due to interference from the thick weathering profile.
- Three downhole petrophysical logs were completed by Gap Geophysics in order to support the 2D seismic surveys.
- An audiomagnetotelluric (AMT) survey was completed across the southeast area of Main deposit, with data processing in progress.
- A gravity ground based survey was designed for mapping basin architecture. The work was suspended at the start of the rain season in 2014 and is planned to recommence in 2015.
- 2,977 soil samples were collected across non-mining areas for multi-element analysis in order to improve understanding of path finder elements for regional exploration.

1.7 **Operations status**

Mining at the Kansanshi operations is currently carried out in two open pits, ie the Main and North West Pits. From the outset and to the end of May 2015, the volume of material mined from these pits was approximately 202 Mbcm and 131 Mbcm, respectively. In 2014, the volume of ore and waste mined was approximately 38 Mbcm, whereas in 2016 it will be approximately 42.5 Mbcm, increasing to over 70 Mbcm from 2018 through to 2023. From 2012 additional mining fleet has been procured to allow the rate of mining to increase to up to 80 Mbcm/annum.

The base of the Main Pit is currently at 1240 mRL (about 180 m below surface), whereas the base of the North West Pit is currently at 1,302.5 mRL. The ultimate depth for the North West Pit (in the Mineral Reserve plan) will be reached in 2022, after which time it can be backfilled with waste.

In 2014, the copper production from all processing circuits (as metal in concentrate and as cathode) was approximately 262 kt, whereas in the year to date the figure is approximately 92 kt. Copper production in the first quarter of 2015 has been constrained in order to limit SXEW plant acid consumption (arising due to the feed of acid-consuming gangue associated with the oxide ore) during the smelter commissioning and ramp-up period. This cathode production constraint will be removed going forward, and the sale of cathode plus concentrate will be replaced with the sale of cathode plus anode product.

Work on the S3 expansion project was placed on hold in 2014, with a commissioning date now revised to mid-2017. From mid-2017 as a first phase of the S3 expansion, the total sulphide capacity will be increased by 12.4 Mtpa. In 2020, upon completion of the S3 expansion, the sulphide capacity will be increased a further 12.6 Mtpa to reach a maximum of 38.2 Mtpa.

In April 2015 the Zambian government announced proposed amendments to the current tax and royalty regime, to be effective from 1st July 2015. Mineral royalties will be set at 9% and corporate tax will be reinstated at 30%, with variable profits tax up to 15%.

1.8 **Environmental status**

More than fifteen environmental impact assessments for operational infrastructure at KMP have been submitted and approved by the Zambia Environmental Management Agency in the last ten years. The environmental and social impacts have been assessed and appropriate mitigation measures have been implemented. The EIAs 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. More recent submissions include the EIA for the smelter, the smelter access road and water storage dams.

The Company is implementing a number of environmental standards in accordance with the ISO 14001 Environmental Management System. The standards provide a structured approach to environmental management including pollution prevention, legal compliance and continued environmental improvement.

No material environmental incident was reported at Kansanshi as at the end of May 2015 and the Company has not been subject to any penalties arising as a result of water pollution or contamination of land beyond the boundaries of its operation. To the Company's knowledge, KMP is not considered by any applicable environmental regulatory authority to be an imminent threat to the environment.

1.9 Mineral Resource summary

Since the December 2012 Technical Report (Titley, CSA, 2012) an additional 332 diamond drilled holes and a further 17,708 grade control reverse circulation (RC) drilled holes were included in this Mineral Resource estimate update. The Mineral Resource estimate update was completed by CSA consultants and the Kansanshi mine geologists under the supervision of the Qualified Person, David Gray.

Combined drillhole data (logging and sampling) together with in-pit geology mapping and observations have facilitated an update of key stratigraphic horizons, the weathering and oxidation profiles and the dominant vein volumes. At the time of this update, only half of the RC chips had been logged but were fully sampled with copper and acid soluble copper grade results. Accordingly, vein volume wireframe models used a combination of data sets and modelling methods to best represent the prevailing vein volumes and orientations. The resultant vein wireframes were volumetrically representative but do not accurately delineate position.

The grade estimate was completed in three stages:

1. Veins were estimated into a 20 m x 20 m x 10 m block model using Length Weighted Inverse Distance Weighting to the power of one (IDW1). Hard boundaries were used between oxidation units. Veins were localised to 5 m x 5 m x 5 m SMUs, to honour the volume estimated by the wireframes.
2. Stratigraphy was estimated using Uniform Conditioning into 40 m x 40 m x 5 m blocks and localised to 10 m x 10 m x 5 m SMUs. Hard boundaries were used between stratigraphic units, while soft boundaries were used between oxidation units in each respective stratigraphic unit.
3. Finally, the proportion of a block that had vein material that had not been wireframed was estimated using Indicator Kriging. The mean grade was assigned to that part of the block that had >50% vein. The final grade of the block comprised the combination of stratigraphy and un-wireframed vein, weighted by their respective proportions.

The wireframes and block model volumes compared well, and were within one percent. Block model grade estimates were validated visually, with summary statistics, comparisons with previous estimates and against mine reconciliation data. Visual validation suggests the grade tenor of the input data is reflected well in the block model. Block grade estimates for copper, acid soluble copper and gold were within 10% of the input drillhole sample grades.

Portions of Main and North West were classified as Measured, Indicated and Inferred, while South East Dome was classified as Indicated and Inferred. Classification was guided by confidence in the geology model and estimation methodology which informs volume, drill hole spacing, QA/QC, and confidence in the grade estimate which was informed by Search Pass and Slope. Wireframes were created for Measured (Search Pass 1 or 2 and Slope>0.7) and Indicated (Search Pass 1 or 2 and Slope > 0.5), and remaining material classified as Inferred. Minor diorite mineralisation was classified as Inferred. Laterite mineralisation is not recoverable and was not classified.

The updated estimates for Kansanshi are presented in the Mineral Resource statement in Table 1-1. The estimate is reported at a 0.2% total copper cut-off grade. Mineral Resources are inclusive of

Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-1 Kansanshi Operations Mineral Resource statement, at May 2015

First Quantum Minerals Limited - Kansanshi Copper Gold Mine Dec 2013 Mineral Resource Estimate depleted to 31st May 2015 Reported at a cut-off grade of 0.2% Tcu						
Classification	**Tonnage (Mt)**	**TCu%**	**Kt Cu**	**ASCu%**	**Au g/t**	**Density**
Measured Resource	105	0.71	738	0.21	0.12	2.44
Indicated Resource	735	0.72	5,313	0.15	0.12	2.61
Total Measured & Indicated	**840**	**0.72**	**6,051**	**0.16**	**0.12**	**2.59**
Inferred Resource	669	0.60	3,993	0.04	0.10	2.75

Mineral Reserve modifying factors have been updated resulting in the marginal copper cut-off grade being reduced from 0.3% to overall averages of 0.3% to 0.2%, depending on plant feed type. Apart from mining depletions, added drillhole data and improved estimation routines, there have been no other changes to the previous 2012 Technical Report Mineral Resource estimates (Titley, CSA, 2012).

1.10 **Mineral Reserve summary**

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

At the outset, conventional Whittle Four-X software was used to determine optimal pit shells for the Main and North West Pits, and for the South East Dome deposit. Optimisations were completed on a maximum net return (NR) basis, and with recoveries to copper metal in concentrate determined from variable and fixed recovery relationships for each deposit. The optimisation process considered pit slope design criteria provided by KMP geotechnical staff, in addition to mining/processing operating costs and metal costs derived and extrapolated in detail from actual costs. Optimisation sensitivity analyses were completed to test the impact of the Zambian government royalty rate increasing from 6% to 9%, and the impact of the Project switching from the sale of cathode plus concentrate product to cathode plus anode.

Despite the obvious cashflow outcome, the imposition of the increased royalty rate has been demonstrated as having neglible impact on the extent to which metal costs contribute to cut-off grade parameters and the Mineral Reserve estimate. Switching from cathode plus concentrate production to cathode plus anode production, also appears to have minimal impact on the cut-off grades and selection of the optimal pit for mine planning.

Following the optimisation, a series of pit designs were developed using the ultimate pit shells, together with groupings of mining stages relating to the current mining areas (as at May 2015) and the progression of these grouped stages as determined from preliminary production schedules.

Detailed LOM production scheduling was then completed to demonstrate an achievable mine plan and hence allow reporting of a Mineral Reserve as stated in Table 1-2. The additional Mineral Reserve within stockpiles (as at May 2015) is stated in Table 1-3. To support the Mineral Reserve estimate, an undiscounted cashflow model has been prepared, inclusive of operating and metal cost revisions (TCRCs) apparent since the time of the optimisation in 2014. This model includes capital costs, but excludes taxation, depreciation and amortisation.

Table 1-2 Kansanshi Operations Mineral Reserve statement, in-pit inventory at May 2015

MINERAL RESERVE AT 31st MAY 2015 (AT $3.00/lb Cu and $1200/ounce Au)							
Leach Ore (Float/leach feed)							
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**	**TCu metal (kt)**	**Au metal (t)**
MAIN & NW	Proved	24.1	1.13	0.71	0.13	273	3.2
	Probable	88.1	1.22	0.76	0.14	1,073	12.5
	Total P+P	**112.1**	**1.20**	**0.75**	**0.14**	**1,346**	**15.7**
SE DOME	Proved						
	Probable	3.8	0.89	0.40	0.14	33	0.5
	Total P+P	**3.8**	**0.89**	**0.40**	**0.14**	**33**	**0.5**
TOTAL	Proved	24.1	1.13	0.71	0.13	273	3.2
	Probable	91.8	1.20	0.75	0.14	1,106	13.0
	Total P+P	**115.9**	**1.19**	**0.74**	**0.14**	**1,379**	**16.2**
Mixed Ore (Float/leach feed)							
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**	**TCu metal (kt)**	**Au metal (t)**
MAIN & NW	Proved	15.3	0.54	0.13	0.09	82	1.4
	Probable	68.1	0.73	0.16	0.13	495	8.5
	Total P+P	**83.4**	**0.69**	**0.15**	**0.12**	**577**	**10.0**
SE DOME	Proved						
	Probable	5.5	0.65	0.15	0.12	36	0.7
	Total P+P	**5.5**	**0.65**	**0.15**	**0.12**	**36**	**0.7**
TOTAL	Proved	15.3	0.54	0.13	0.09	82	1.4
	Probable	73.6	0.72	0.16	0.12	530	9.2
	Total P+P	**88.9**	**0.69**	**0.15**	**0.12**	**613**	**10.6**
Sulphide Ore (Float feed)							
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**	**TCu metal (kt)**	**Au metal (t)**
MAIN & NW	Proved	71.2	0.49	0.02	0.11	352	7.6
	Probable	345.3	0.57	0.02	0.12	1,969	40.6
	Total P+P	**416.5**	**0.56**	**0.02**	**0.12**	**2,321**	**48.3**
SE DOME	Proved						
	Probable	46.8	0.80	0.03	0.13	372	6.2
	Total P+P	**46.8**	**0.80**	**0.03**	**0.13**	**372**	**6.2**
TOTAL	Proved	71.2	0.49	0.02	0.11	352	7.6
	Probable	392.1	0.60	0.02	0.12	2,341	46.8
	Total P+P	**463.2**	**0.58**	**0.02**	**0.12**	**2,693**	**54.4**
Total Ore							
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**	**TCu metal (kt)**	**Au metal (t)**
MAIN & NW	Proved	110.6	0.64	0.18	0.11	707	12.3
	Probable	501.4	0.71	0.17	0.12	3,537	61.7
	Total P+P	**612.0**	**0.69**	**0.17**	**0.12**	**4,244**	**73.9**
SE DOME	Proved						
	Probable	56.0	0.79	0.06	0.13	441	7.3
	Total P+P	**56.0**	**0.79**	**0.06**	**0.13**	**441**	**7.3**
TOTAL	Proved	110.6	0.64	0.18	0.11	707	12.3
	Probable	**557.4**	**0.71**	**0.16**	**0.12**	3,978	69.0
	Total P+P	**668.0**	**0.70**	**0.16**	**0.12**	**4,685**	**81.3**

Table 1-3 Kansanshi Operations Mineral Reserve statement, stockpile inventory at May 2015

MINERAL RESERVE AT 31st DECEMBER 2015 (AT $3.00/lb Cu and $1200/ounce Au)							
Leach Ore (Float/leach feed)							
S/Piles	Class	Mtonnes	TCu (%)	ASCu (%)	Au (g/t)	TCu metal (kt)	Au metal (t)
TOTAL	Proved						
	Probable	0.0	0.00	0.00		0	
	Total P+P	**0.0**	**0.00**	**0.00**		**0**	
Mixed Ore (Float/leach feed)							
S/Piles	Class	Mtonnes	TCu (%)	ASCu (%)	Au (g/t)	TCu metal (kt)	Au metal (t)
TOTAL	Proved						
	Probable	46.4	0.55	0.23		256	0.0
	Total P+P	**46.4**	**0.55**	**0.23**		**256**	
Sulphide Ore (Float feed)							
S/Piles	Class	Mtonnes	TCu (%)	ASCu (%)	Au (g/t)	TCu metal (kt)	Au metal (t)
TOTAL	Proved						
	Probable	7.2	0.36	0.02		26	0.0
	Total P+P	**7.2**	**0.36**	**0.02**		**26**	
Total Ore							
S/Piles	Class	Mtonnes	TCu (%)	ASCu (%)	Au (g/t)	TCu metal (kt)	Au metal (t)
TOTAL	Proved						
	Probable	53.6	0.53	0.20		282	0.0
	Total P+P	**53.6**	**0.53**	0.20		282	0.0

1.11 Production schedule

Table 1-4 summarises the LOM production schedule, whilst Figure 1-1 shows the LOM mining sequence. Features of the LOM mining and production schedule associated with the detailed pit designs are as follows:

- The production shown for 2015 represents that for the remaining seven months of the year. From 2016 the remaining Project life is 16 years (ie, to 2031).
- The total material mined from all pits amounts to 2,119.2 Mt (845.2 Mbcm), of which 668.0 Mt is ore (including mixed-leach, mixed-float and sulphide ore) and 1,451.2 Mt is waste (including refractory and inferred ore). 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%.
- The direct feed float/leach ore is 134.5 Mt at a grade of 1.18%Cu and 46.4 Mt at a grade of 0.55% Cu is stockpile reclaim from existing stockpiles.
- The total float/leach ore mined to stockpiles and reclaimed throughout the Project life is 70.3 Mt at a grade of 0.60% Cu.
- The direct feed sulphide ore is 354.8 Mt at a grade of 0.67%Cu and 7.2 Mt at a grade of 0.36% Cu is stockpile reclaim from existing stockpiles.
- The total sulphide ore mined to stockpiles and reclaimed throughout the Project life is 108.4 Mt at a grade of 0.30% Cu.
- The current plant configuration has a capacity of 15.6 Mtpa of float/leach feed and 13.2 Mtpa of sulphide feed. In mid-2017 as a first phase of the S3 expansion, the total sulphide capacity

is increased by 12.4 Mtpa. In 2020, upon completion of the S3 expansion, the sulphide capacity is increased a further 12.6 Mtpa to reach a maximum of 38.2 Mtpa.

- The average copper grade for float/leach feed between 2015 and 2019 is 1.20% Cu. The average copper grade for sulphide feed during the same period is 0.88% Cu. Thereafter the average feed grades drop to 0.86% Cu in float/leach and 0.57% Cu in sulphide over the rest of the LOM period.
- Current annual copper metal production is 250,000 tonnes. Between 2017 and 2027, the average annual copper metal production is 305,000 tonnes (on a diluted feed basis), before tailing off in the final years of production.
- The annual average gold production is approximately 104 thousand ounces (on a diluted feed basis).
- The overall life of mine strip ratio (tonnes) is 2.16 : 1.
- Main Pit is mined extensively throughout the life of the mine and has three production areas. A central area, which is already stripped of waste and ore, is exposed and currently being mined. The other areas are phase 1 push back areas which are the next areas designated for pre-strip mining, and the ultimate pushbacks to achieve the final pit configuration.
- The North West Pit is depleted by 2019 in the schedule and thereafter is available for inpit backfill of waste from the Main Pit pushback areas.
- The South East Dome Pit is mined from 2017 to 2023 at an average annual mining rate of around 40 Mt.
- The current stockpiles are rehandled between 2016 and 2027 at an average rate of around 5 Mtpa. The maximum float-leach stockpile size of 81.6 Mt is reached in 2022, whereas the maximum sulphide stockpile of size 76.4 Mt is reached in 2023.

1.12 **Capital and operating costs**

Table 1-5 lists the capital costs included in the Project cashflow model. An additional $90.7 M is included at the end of the Project life to cover closure costs. A detailed review of actual operating costs was completed for the optimisation work carried out in 2014, yielding summary figures as follows:

- mining waste = $3.13/t waste (overall average for all pits)
- mining ore = $3.00/t ore (overall average for all pits)
- processing costs = $13.54/t for mixed leach ore, $10.82/t for mixed float ore and $8.88/t for sulphide ore

The metal costs (treatment charges, refining charges (TCRCs)) were also determined from a review of actual costs. In the 2014 optimisation, the metal costs reflected the sale of concentrate and also a 6% royalty, yielding summary figures as follows:

- total cathode copper metal costs = $0.36/lb Cu
- total copper in concentrate metal costs = $0.36/lb
- total gold in copper concentrate metal costs = $556.35/oz for mixed feed and $418.03/oz for sulphide feed (based on overall average head grades and copper recovery values)
- total gold in gravity circuit metal costs = $92.00/oz

Table 1-4 Kansanshi Operations life of mine production schedule

	UNITS		TOTAL	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
MINING																				
Total ore (incl. mining dil'n/rec.)	Tonnes	Mt	**668.0**	30.7	60.2	41.5	63.3	56.9	51.9	53.3	58.5	61.6	36.9	33.9	45.8	34.6	23.0	10.7	4.1	0.9
	Cu	%	**0.71**	0.88	0.88	0.82	0.78	0.76	0.74	0.71	0.73	0.62	0.68	0.61	0.50	0.54	0.55	0.46	0.56	0.69
Total waste (incl. mining dil'n/rec.)	Tonnes	Mt	**1,451.2**	82.1	97.7	130.1	130.6	120.4	143.0	141.6	128.3	133.3	93.5	69.0	84.9	50.8	29.4	13.0	3.1	0.4
Strip ratio	W : O	t: t	**2.17**	2.67	1.62	3.14	2.06	2.12	2.75	2.65	2.19	2.16	2.53	2.04	1.85	1.47	1.28	1.21	0.75	0.43
Direct feed (incl. mining dil'n/rec.)	Tonnes	Mt	**489.3**	15.9	26.60	28.94	33.83	35.74	46.22	45.24	49.06	42.34	25.97	31.49	38.44	33.43	20.87	10.65	3.72	0.86
	Cu	%	**0.81**	1.14	1.00	1.06	1.09	1.05	0.80	0.79	0.81	0.77	0.86	0.63	0.55	0.55	0.58	0.47	0.59	0.73
Stockpile on (incl. mining dil'n/rec.)	Tonnes	Mt	**178.8**	14.9	33.59	12.54	29.50	21.14	5.71	8.09	9.48	19.28	10.94	2.38	7.37	1.22	2.12	0.09	0.36	0.08
	Cu	%	**0.43**	0.61	0.78	0.26	0.43	0.27	0.27	0.28	0.33	0.29	0.26	0.32	0.23	0.34	0.27	0.23	0.23	0.23
Stockpile off (incl. mining dil'n/rec.)	Tonnes	Mt	**232.4**	0.0	2.13	5.98	7.26	5.36	7.45	8.42	4.60	11.32	27.70	22.18	15.23	20.23	32.79	37.11	17.01	7.60
	Cu	%	**0.44**	0.00	1.21	0.66	0.46	0.46	0.54	0.53	0.50	0.50	0.45	0.37	0.52	0.47	0.34	0.35	0.54	0.28
Stockpile Balance	Tonnes	Mt		68.5	99.9	106.5	128.7	144.5	142.8	142.5	147.3	155.3	138.5	118.8	110.9	91.9	61.2	24.2	7.5	0.0
	Cu	%		0.54	0.61	0.57	0.54	0.50	0.49	0.48	0.47	0.44	0.43	0.44	0.41	0.40	0.43	0.54	0.54	0.00
TOTAL FEED TO PLANT (after mining dil'n & recovery)																				
Mixed float/leach	Tonnes	Mt	**251.2**	9.9	15.6	15.6	15.6	15.6	15.6	15.6	15.6	15.6	15.6	15.6	15.6	15.6	15.6	15.6	15.6	7.9
	Cu	%	**0.89**	1.27	1.15	1.18	1.21	1.20	0.95	0.89	1.02	0.86	0.94	0.80	0.76	0.68	0.63	0.59	0.60	0.30
	Au	g/t	**0.12**	0.14	0.15	0.12	0.16	0.09	0.12	0.10	0.11	0.12	0.12	0.13	0.12	0.13	0.12	0.13	0.13	0.13
Sulphide float	Tonnes	Mt	**470.4**	6.0	13.2	19.4	25.5	25.5	38.1	38.1	38.1	38.1	38.1	38.1	38.1	38.1	38.1	32.2	5.2	0.5
	Cu	%	**0.58**	0.91	0.87	0.84	0.84	0.83	0.69	0.69	0.69	0.65	0.53	0.41	0.45	0.45	0.35	0.27	0.38	0.59
	Au	g/t	**0.12**	0.09	0.13	0.13	0.16	0.13	0.12	0.12	0.12	0.10	0.11	0.10	0.09	0.11	0.11	0.12	0.12	0.24
Total feed	**Tonnes**	**Mt**	**721.6**	**15.9**	**28.7**	**34.9**	**41.1**	**41.1**	**53.7**	**53.7**	**53.7**	**53.7**	**53.7**	**53.7**	**53.7**	**53.7**	**53.7**	**47.8**	**20.7**	**8.5**
	Cu	**%**	**0.69**	**1.14**	**1.02**	**0.99**	**0.98**	**0.97**	**0.76**	**0.75**	**0.78**	**0.71**	**0.65**	**0.52**	**0.54**	**0.52**	**0.43**	**0.38**	**0.55**	**0.32**
	Au	**g/t**	**0.12**	**0.12**	**0.14**	**0.13**	**0.16**	**0.11**	**0.12**	**0.11**	**0.12**	**0.10**	**0.11**	**0.11**	**0.10**	**0.12**	**0.11**	**0.13**	**0.13**	**0.14**
AVERAGE RECOVERIES																				
Mixed float/leach	Cu	%	**77.9%**	79.8%	79.3%	79.0%	78.0%	79.5%	77.4%	77.3%	78.4%	78.4%	78.0%	76.9%	77.5%	79.5%	62.7%	79.8%	79.7%	79.7%
	Au	%	**66.2%**	65.3%	65.6%	65.5%	65.9%	65.9%	66.3%	66.3%	66.2%	66.4%	66.4%	66.5%	66.5%	66.3%	66.6%	66.4%	66.4%	67.4%
Sulphide float	Cu	%	**89.4%**	87.9%	88.5%	89.4%	86.6%	88.6%	90.1%	89.9%	89.8%	89.8%	88.9%	89.2%	90.1%	90.5%	90.0%	89.8%	89.5%	88.4%
	Au	%	**63.4%**	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%
Overal average	Cu	%	**84.1%**	**82.4%**	**83.3%**	**83.8%**	**82.4%**	**84.1%**	**85.0%**	**85.5%**	**85.9%**	**85.5%**	**84.3%**	**83.4%**	**84.8%**	**86.8%**	**78.2%**	**84.8%**	**81.3%**	**80.8%**
	Au	%	**64.4%**	**64.8%**	**64.6%**	**64.3%**	**64.4%**	**64.2%**	**64.3%**	**64.1%**	**64.2%**	**64.4%**	**64.4%**	**64.5%**	**64.5%**	**64.4%**	**64.4%**	**64.4%**	**65.7%**	**67.0%**
METAL RECOVERED (after mining dil'n & recovery)																				
Mixed float/leach	Cu	kt	**1,749.8**	100.4	142.4	144.3	147.6	149.0	114.2	107.7	124.7	104.9	114.3	95.9	91.5	84.4	61.7	73.3	74.3	19.2
	Au	koz	**666.5**	28.7	47.9	40.0	51.5	29.9	40.5	32.5	37.4	39.0	41.1	43.1	40.0	43.5	41.2	44.1	43.1	23.0
Sulphide float	Cu	kt	**2,443.9**	48.3	101.5	145.9	184.2	187.3	234.6	236.1	236.7	221.7	178.8	138.9	154.6	156.5	119.6	79.0	17.6	2.8
	Au	koz	**1,105.7**	11.5	35.1	52.0	81.2	66.7	93.8	93.5	92.2	75.8	86.2	79.0	72.7	86.9	84.3	79.9	12.2	2.7
Total recovered	Cu	kt	**4,193.7**	148.8	243.8	290.2	331.8	336.3	348.8	343.7	361.4	326.6	293.0	234.8	246.0	240.9	181.3	152.3	91.9	22.1
	Au	koz	**1,772.2**	40.2	83.1	92.0	132.7	96.5	134.3	126.0	129.6	114.8	127.3	122.1	112.6	130.4	125.5	124.0	55.3	25.7

Figure 1-1 LOM schedule – mining sequence

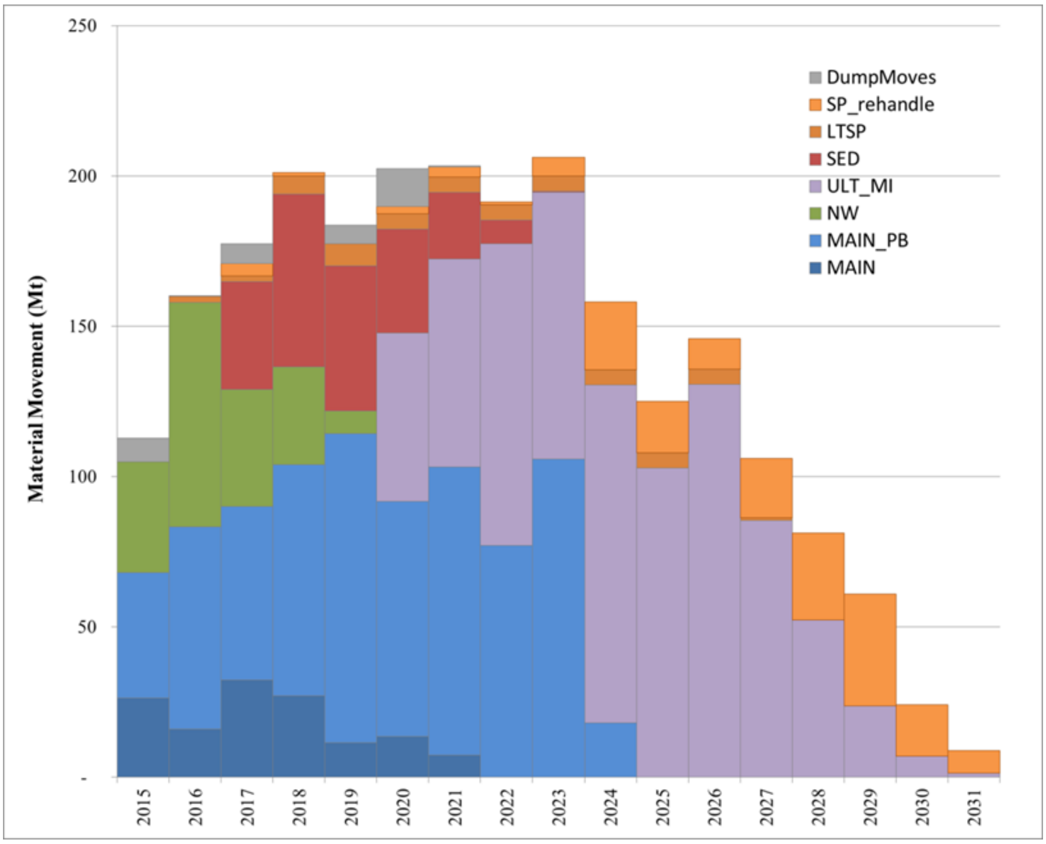


Table 1-5 Capital costs, 2015 to 2020

Capital Cost Items ($M)	2015	2016	2017	2018	2019	2020	Total
Operations							
Mining	34.55	29.81	12.29	8.78	-	-	85.43
Process	6.40	11.40	4.20	10.15	0.57	-	32.72
Engineering	2.98	22.97	2.80	0.10	0.43	-	29.27
Resource optimisation	0.72	0.10	0.10	0.10	0.10	-	1.12
Security	0.25	-	0.15	-	-	-	0.40
Safety	0.21	0.08	0.08	0.08	0.08	-	0.51
Commercial	0.28	0.25	0.25	0.25	0.25	-	1.28
Enviroment	0.57	0.25	0.25	0.25	0.25	-	1.57
IT	0.02	0.20	0.20	0.20	0.20	-	0.82
PR	-	0.01	0.01	0.01	0.01	-	0.04
Construction	3.06	-	-	-	-	-	3.06
Finance	0.25	-	-	-	-	-	0.25
Exploration	-	-	-	-	-	-	-
FQMO Roads projects	8.35	12.20	0.20	0.50	-	-	21.25
Subtotal	**57.63**	**77.27**	**20.52**	**20.41**	**1.88**	**-**	**177.71**
Projects							
Kansanshi smelter	(0.05)	-	-	-	-	-	(0.05)
Kansanshi smelter phase 2	85.80	100.62	134.16	167.70	134.16	48.36	670.80
Kansanshi smelter phase2 ISACONVERT	12.92	32.08	-	-	-	-	45.00
S3 expansion project	15.73	100.00	230.00	80.45	26.82	-	453.00
Power lines project	42.84	-	-	-	-	-	42.84
New plant workshop	3.59	-	-	-	-	-	3.59
Mine workshop extension	0.84	-	-	-	-	-	0.84
Other Kansanshi	11.90	15.00	-	-	-	-	26.90
Subtotal	**173.59**	**247.70**	**364.16**	**248.15**	**160.98**	**48.36**	**1,242.93**
Other							
Developtment Costs (Mining)	-	-	-	-	-	-	-
Smelter Plant - Ramp Up Costs	-	-	-	-	-	-	-
Total	**231.22**	**324.96**	**384.69**	**268.56**	**162.86**	**48.36**	**1,420.64**

The 2014 optimisation input costs were reviewed in 2015, especially in relation to the metal costs. The TCRCs were amended to reflect the sale of anode and the royalty payment rate was adjusted to 9%. This yielded the following revised figures:

- total cathode copper metal costs = $0.45/lb Cu
- total copper in anode metal costs = $0.59/lb Cu
- average float/leach feed metal costs = $0.54/lb Cu
- total gold in copper concentrate metal costs = $247.20/oz for float/leach feed and for sulphide float feed (based on overall average head grades and copper recovery values)
- total gold in gravity circuit metal costs = $128.00/oz

1.13 **Economic analysis**

An economic analysis in the form of an undiscounted cashflow model to support the Mineral Reserve estimate is listed in Table 1-6. This model shows the indicative cashflow and does not replace a more comprehensive Project financial model inclusive of taxation, depreciation and amortisation, and from which an accurate NPV and IRR can be calculated.

The annual revenues are calculated from the same metal prices as used in the pit optimisation process (Item 15):

- Copper = US$3.00/lb (US$6,615/t)
- Molybdenum = US$13.50/lb ($US29,762/t)
- Gold = US$1,200/oz
- Silver = US$16.00/oz

Other important analysis parameters are:

- mining waste = $3.13/t waste (overall average for all pits)
- mining ore = $3.00/t ore (overall average for all pits)
- processing costs = $13.54/t for mixed leach ore, $10.82/t for mixed float ore and $8.88/t for sulphide ore
- stockpile reclaim costs = $1.30/t
- total cathode copper metal costs = $0.45/lb Cu
- total copper in anode metal costs = $0.59/lb Cu
- average float/leach feed metal costs = $0.54/lb Cu
- total gold in copper concentrate metal costs = $247.20/oz float/leach feed and for sulphide float feed (based on overall average head grades and copper recovery values)
- total gold in gravity circuit metal costs = $128.00/oz

The recovery values shown in Table 22-1 1-6 are average figures resulting from the application, on a mining model block by block basis, of the variable relationships listed in Item 15. The cashflow for 2015, shows the partial year revenue and operating expenditure (from June), but the full year capital expenditure.

Table 1-6 Mineral Reserves cashflow model summary

	UNITS		TOTAL	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
MINING																				
Total ore	Tonnes	Mt	**668.03**	30.75	60.19	41.47	63.33	56.88	51.93	53.34	58.54	61.62	36.91	33.87	45.81	34.65	23.00	10.74	4.07	0.93
Total waste	Tonnes	Mt	**1,451.18**	82.11	97.68	130.05	130.63	120.38	143.01	141.60	128.28	133.32	93.53	69.04	84.92	50.76	29.42	13.02	3.06	0.40
Strip ratio			**2.17**	2.67	1.62	3.14	2.06	2.12	2.75	2.65	2.19	2.16	2.53	2.04	1.85	1.47	1.28	1.21	0.75	0.43
Total reclaim	**Tonnes**	**t**	**232.38**	**0.00**	**2.13**	**5.98**	**7.26**	**5.36**	**7.45**	**8.42**	**4.60**	**11.32**	**27.70**	**22.18**	**15.23**	**20.23**	**32.79**	**37.11**	**17.01**	**7.60**
TOTAL FEED TO PLANT (after mining dil'n & recovery)																				
Mixed leach/float	Tonnes	Mt	**251.21**	9.88	15.56	15.56	15.56	15.56	15.56	15.56	15.56	15.56	15.56	15.56	15.56	15.56	15.56	15.56	15.56	7.92
	Cu	%	**0.89**	1.27	1.15	1.18	1.21	1.20	0.95	0.89	1.02	0.86	0.94	0.80	0.76	0.68	0.63	0.59	0.60	0.30
	Au	g/t	**0.12**	0.14	0.15	0.12	0.16	0.09	0.12	0.10	0.11	0.12	0.12	0.13	0.12	0.13	0.12	0.13	0.13	0.13
Sulphide float	Tonnes	Mt	**470.43**	5.99	13.17	19.35	25.54	25.54	38.10	38.10	38.10	38.10	38.10	38.10	38.10	38.10	38.10	32.20	5.17	0.54
	Cu	%	**0.58**	0.91	0.87	0.84	0.84	0.83	0.69	0.69	0.69	0.65	0.53	0.41	0.45	0.45	0.35	0.27	0.38	0.59
	Au	g/t	**0.12**	0.09	0.13	0.13	0.16	0.13	0.12	0.12	0.12	0.10	0.11	0.10	0.09	0.11	0.11	0.12	0.12	0.24
Total feed	**Tonnes**	**Mt**	**721.63**	**15.86**	**28.73**	**34.91**	**41.10**	**41.10**	**53.67**	**53.67**	**53.67**	**53.67**	**53.67**	**53.67**	**53.67**	**53.67**	**53.67**	**47.77**	**20.73**	**8.45**
	Cu	**%**	**0.69**	**1.14**	**1.02**	**0.99**	**0.98**	**0.97**	**0.76**	**0.75**	**0.78**	**0.71**	**0.65**	**0.52**	**0.54**	**0.52**	**0.43**	**0.38**	**0.55**	**0.32**
	Au	**g/t**	**0.12**	**0.12**	**0.14**	**0.13**	**0.16**	**0.11**	**0.12**	**0.11**	**0.12**	**0.10**	**0.11**	**0.11**	**0.10**	**0.12**	**0.11**	**0.13**	**0.13**	**0.14**
AVERAGE RECOVERIES																				
Mixed float/leach	Cu	%	**77.9%**	79.8%	79.3%	79.0%	78.0%	79.5%	77.4%	77.3%	78.4%	78.4%	78.0%	76.9%	77.5%	79.5%	62.7%	79.8%	79.7%	79.7%
	Au	%	**66.2%**	65.3%	65.6%	65.5%	65.9%	65.9%	66.3%	66.3%	66.2%	66.4%	66.4%	66.5%	66.5%	66.3%	66.6%	66.4%	66.4%	67.4%
Sulphide float	Cu	%	**89.4%**	87.9%	88.5%	89.4%	86.6%	88.6%	90.1%	89.9%	89.8%	89.8%	88.9%	89.2%	90.1%	90.5%	90.0%	89.8%	89.5%	88.4%
	Au	%	**63.4%**	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%
Overal average	Cu	%	**84.1%**	**82.4%**	**83.3%**	**83.8%**	**82.4%**	**84.1%**	**85.0%**	**85.5%**	**85.9%**	**85.5%**	**84.3%**	**83.4%**	**84.8%**	**86.8%**	**78.2%**	**84.8%**	**81.3%**	**80.8%**
	Au	%	**64.4%**	**64.8%**	**64.6%**	**64.3%**	**64.4%**	**64.2%**	**64.3%**	**64.1%**	**64.2%**	**64.4%**	**64.4%**	**64.5%**	**64.5%**	**64.4%**	**64.4%**	**64.4%**	**65.7%**	**67.0%**
METAL RECOVERED (after mining dil'n & recovery)																				
Mixed float/leach	Cu	kt	**1,749.8**	100.4	142.4	144.3	147.6	149.0	114.2	107.7	124.7	104.9	114.3	95.9	91.5	84.4	61.7	73.3	74.3	19.2
	Au	koz	**666.5**	28.7	47.9	40.0	51.5	29.9	40.5	32.5	37.4	39.0	41.1	43.1	40.0	43.5	41.2	44.1	43.1	23.0
Sulphide float	Cu	kt	**2,443.9**	48.3	101.5	145.9	184.2	187.3	234.6	236.1	236.7	221.7	178.8	138.9	154.6	156.5	119.6	79.0	17.6	2.8
	Au	koz	**1,105.7**	11.5	35.1	52.0	81.2	66.7	93.8	93.5	92.2	75.8	86.2	79.0	72.7	86.9	84.3	79.9	12.2	2.7
Total recovered	Cu	kt	**4,193.7**	148.8	243.8	290.2	331.8	336.3	348.8	343.7	361.4	326.6	293.0	234.8	246.0	240.9	181.3	152.3	91.9	22.1
	Au	koz	**1,772.2**	40.2	83.1	92.0	132.7	96.5	134.3	126.0	129.6	114.8	127.3	122.1	112.6	130.4	125.5	124.0	55.3	25.7
GROSS REVENUE																				
Cu $/t	6,613.86	$M	**27,736.4**	983.8	1,612.8	1,919.2	2,194.5	2,224.3	2,307.1	2,273.3	2,390.3	2,159.8	1,938.2	1,552.8	1,627.2	1,593.3	1,199.0	1,007.1	607.7	145.9
Au $/oz	1,200.00	$M	**2,126.7**	48.2	99.7	110.4	159.3	115.8	161.1	151.2	155.5	137.8	152.8	146.5	135.2	156.5	150.6	148.8	66.4	30.8
subtotal		**$M**	**29,863.1**	**1,032.1**	**1,712.5**	**2,029.6**	**2,353.8**	**2,340.2**	**2,468.2**	**2,424.6**	**2,545.8**	**2,297.6**	**2,090.9**	**1,699.3**	**1,762.4**	**1,749.7**	**1,349.6**	**1,155.9**	**674.1**	**176.7**
CAPITAL COSTS																				
Operations capital		$M	**177.7**	57.6	77.3	20.5	20.4	1.9	0.0											
Projects capital		$M	**1,242.9**	173.6	247.7	364.2	248.2	161.0	48.4											
Sustaining capex (5% of total opex)		$M	**703.9**	26.5	40.2	45.5	51.4	48.7	57.0	57.0	55.5	57.1	48.2	43.5	47.4	40.8	36.3	29.7	13.8	5.3
Closure and reclamation		$M	**90.7**															30.2	30.2	30.2
subtotal		**$M**	**2,215.2**	**257.7**	**365.1**	**430.1**	**320.0**	**211.6**	**105.3**	**57.0**	**55.5**	**57.1**	**48.2**	**43.5**	**47.4**	**40.8**	**36.3**	**59.9**	**44.0**	**35.6**
OPERATING COSTS																				
Mining																				
Ore $/t mined	3.00	$M	**2,004.1**	92.2	180.6	124.4	190.0	170.7	155.8	160.0	175.6	184.9	110.7	101.6	137.4	103.9	69.0	32.2	12.2	2.8
Waste $/mined	3.13	$M	**4,542.2**	257.0	305.7	407.1	408.9	376.8	447.6	443.2	401.5	417.3	292.8	216.1	265.8	158.9	92.1	40.7	9.6	1.2
Processing (incl G&A)																				
Mixed-leach $/t process	13.54	$M	**1,573.4**	100.0	141.5	143.0	116.5	121.0	90.7	91.0	97.1	83.4	82.9	79.8	81.7	90.0	74.2	84.3	86.2	10.2
Mixed-float $/t process	10.82	$M	**1,467.5**	27.3	55.8	54.5	75.7	72.1	96.3	96.1	91.2	102.1	102.6	105.0	103.5	96.9	109.5	101.4	99.9	77.7
Sulphide-float $/t process	8.88	$M	**4,188.6**	53.3	117.2	172.3	227.4	227.4	339.3	339.3	339.3	339.3	339.3	339.3	339.3	339.3	339.3	286.7	46.0	4.8
Stockpile reclaim $/t ore	1.30	$M	**302.1**	0.0	2.8	7.8	9.4	7.0	9.7	10.9	6.0	14.7	36.0	28.8	19.8	26.3	42.6	48.2	22.1	9.9
subtotal		**$M**	**14,078.0**	**529.7**	**803.5**	**909.1**	**1,027.9**	**974.9**	**1,139.4**	**1,140.5**	**1,110.7**	**1,141.7**	**964.2**	**870.6**	**947.5**	**815.3**	**726.6**	**593.7**	**276.0**	**106.6**
METAL COSTS (INCLUDING 9% ROYALTIES)																				
Mixed float/leach																				
Total Cu Metal Cost		$M	**2,066.1**	118.6	168.1	170.4	174.3	175.9	134.9	127.1	147.3	123.8	135.0	113.2	108.0	99.7	72.8	86.6	87.7	22.7
Total Au Metal Cost		$M	**332.5**	18.0	26.2	25.7	27.5	3.5	21.5	19.5	22.5	19.8	21.4	19.0	18.0	17.4	13.8	15.8	15.8	5.5
subtotal		**$M**	**2,398.7**	**136.5**	**194.4**	**196.1**	**201.8**	**179.4**	**156.4**	**146.6**	**169.8**	**143.6**	**156.4**	**132.2**	**126.0**	**117.1**	**86.6**	**102.4**	**103.6**	**28.2**
subtotal (excluding royalties)		**$M**	**1,149.5**	**67.1**	**95.2**	**96.5**	**98.7**	**77.8**	**76.5**	**72.0**	**83.4**	**70.2**	**76.5**	**64.2**	**61.3**	**56.6**	**41.4**	**49.2**	**49.9**	**13.0**
Sulphide float																				
Total Cu Metal Cost		$M	**3,204.0**	63.4	133.0	191.3	241.5	245.6	307.5	309.5	310.3	290.7	234.3	182.1	202.6	205.1	156.8	103.5	23.1	3.7
Total Au Metal Cost		$M	**480.5**	8.3	18.7	27.1	36.0	34.9	44.9	44.9	44.6	41.0	35.8	29.2	30.8	32.3	26.9	20.4	4.0	0.7
subtotal		**$M**	**3,684.5**	**71.6**	**151.7**	**218.3**	**277.5**	**280.5**	**352.4**	**354.4**	**354.8**	**331.6**	**270.1**	**211.3**	**233.4**	**237.5**	**183.8**	**124.0**	**27.0**	**4.4**
subtotal (excluding royalties)		**$M**	**2,110.3**	**41.6**	**87.5**	**125.9**	**159.1**	**161.8**	**202.6**	**203.8**	**204.0**	**191.5**	**154.4**	**120.1**	**133.5**	**135.0**	**103.5**	**68.3**	**15.2**	**2.4**
CASHFLOW		**$M**	**7,486.7**	**36.4**	**197.8**	**275.9**	**526.6**	**693.8**	**714.7**	**726.0**	**854.9**	**623.6**	**652.0**	**441.6**	**408.2**	**539.2**	**316.3**	**276.0**	**223.5**	**1.9**

1.14 **Conclusions and recommendations**

1.14.1 **Mineral Resource**

The drillhole dataset used in this updated Mineral Resource estimate is extensive in its coverage of the Kansanshi deposits. QAQC of samples demonstrates representative assay results for both diamond and the last two years of RC drilled samples. Geological understanding and resulting models are good and conform with the extensive deposit geology exposed in current pits. The large number of close spaced RC holes adds confidence to local geology and grade continuity of the respective domains of mineralisation. The Qualified Person, David Gray believes this updated Mineral Resource estimate to be representative of the prevailing geology and drilled sample data.

Recommendations in respect of the Mineral Resource estimate are as follows:

1. Should pre-2013 RC grade control data be used in future estimates, bias and precision risks will need to be addressed through relevant comparisons with current RC and diamond drilling/sampling/assaying data that is supported by robust QAQC results.
2. Current grade control RC sample QAQC must continue in order to maximise value and quality of sample assay results in addition to assuring accurate grade control model estimates and mark out for mining.
3. Duplication of collars, sample bias, and poor recoveries in solution cavities should be reviewed and procedures adjusted where necessary.
4. Surfaces for the stratigraphic units are treated as sub-horizontal and conformable. However, in-pit mapping is noted for its complex structural deformation, which, if resolved, and at a relevant scale, will need to be considered in future geology models.
5. Continued geological logging of grade control RC chips will improve vein volume modelling and estimates.

1.14.2 **Mineral Reserve**

In the opinion of Michael Lawlor (QP), the Mineral Reserve estimate reflects an achievable mining plan and production sequence, and one which has taken account of open pit mining phases and optimised processing plant feed delivery.

There is considered to be minimal risk attributable to the mining method and to the primary equipment that has been adopted and remains in use for the Operation.

Uncertainty in operating costs, to the extent identified in optimisation sensitivity analyses, poses little risk to the selection of the optimal pit shells as the basis for all following pit design and production scheduling work supporting the Mineral Reserve estimate. Revisions to the metal costs (TCRCs) also do not significantly impact the selection of the optimal shells, but have nevertheless, been included into the supporting cashflow model.

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

1. As part of a business intelligence system currently being developed for KMP, it is recommended that operating costs continue to be reviewed for future mine planning and that use be made of this system to facilitate the process.

2. It is recommended that periodic reviews be undertaken to account for the prevailing and projected market conditions and their impact on future mine planning.
3. Variable processing recovery relationships need to be periodically reviewed in the light of continuing processing operations, and updates then incorporated into future optimisations and modelling.
4. As part of the normal cyclic development of operating mine plans and mine design updates, the designs shown in this Technical Report should be reviewed against the detailed geotechnical criteria specified by KMP.

ITEM 2 **INTRODUCTION**

2.1 Purpose of this report

This Technical Report on the Kansanshi Operations (the property) has been prepared by Qualified Persons David Gray, Michael Lawlor and Robert Stone of First Quantum Minerals Pty Ltd (FQM, the issuer) for FQM. The purpose of this Technical Report is to document updated Mineral Resource and Mineral Reserve estimates for the property, together with supporting information on Project cost estimates and Project development status.

2.2 Terms of reference

This Technical Report covers the Main, North West and South East Dome deposits at the Kansanshi Operations and has been written to comply with the reporting requirements of the Canadian National Instrument 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, Australian Institute of Geoscientists and 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. Mr Malcolm Titley of CSA and Mr Gray are joint authors of Items 7 to 12 and Item 14. 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 Robert Stone (QP). Mr Stone 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 responsibility of each QP.

Name	Position	NI 43-101 Contribution
David Gray *BSc(Geology), MAusIMM, PrSciNat(SACNASP)*	Group Mine and Resource Geologist, FQM (Australia) Pty Ltd	Author and Qualified Person Items 7– 12, 14
Michael Lawlor *BEng Hons (Mining), MEngSc, FAusIMM*	Consultant Mining Engineer, FQM (Australia) Pty Ltd	Author and Qualified Person Items 1 to 6, 15 and 16, 19 to 26
Robert Stone *BSc(Hons), CEng, ACSM*	Technical Manager, FQM (Australia) Pty Ltd	Author and Qualified Person Items 13, 17 and 18
Malcolm Titley *BSc, MAusIMM, MAIG*	Principal CSA Global (UK) Ltd	Contributing Author Items 7 – 12, 14

2.4 **Sources of information**

Information used in compiling this Technical Report was derived from previous technical reports on the property, and from the reports and documents listed in the References (Item 27). In particular, repeated reference is made to the two Technical Reports filed under an FQM covering letter in December 2012, ie:

- Journet, N. & Cameron, T., DumpSolver, 2012. *Technical Report on the Updated Mineral Reserves, Kansanshi Copper Mine, North West Province, Zambia. Open Pit Optimisation, Pit Design and Mineral Reserves.* Report to First Quantum Minerals Ltd by DumpSolver Pty Ltd, December.
- Titley, M., CSA, 2012. *NI 43-101 Technical Report on the Kansanshi and South East Dome Mineral Resource Estimate Update, North West Province, Zambia, Africa.* Report to First Quantum Minerals Ltd by CSA Global (UK) Ltd, December 2012.
- Titley, M, CSA, 2014. *R106 2014 FQM_Kansanshi 2013 MRE_Summary Report.* Internal report to First Quantum Minerals Ltd by CSA Global (UK) Ltd, April 2014.

In respect of the Mineral Resource estimates, quality control procedures and internal and external quality control data were provided by FQM and reviewed by CSA. The following key data was also provided to CSA for use in the estimates:

- drill hole SQL database, containing collar location, downhole survey, assay and geology data
- bulk density data
- 3-dimensional wireframes for stratigraphic boundaries, geological units, veins, weathering and the mineralogical oxidation profiles
- topography wireframe
- depletion pit surfaces which model the base of mining at the end of May 2015
- deposit area wireframes which divide the Mineral Resource model into three deposits
- powerpoint presentations by FQM (FQM 2012[1,2] to FQM 2013[1],[2,3]) summarising:
    - grade control QC data
    - oxidation domain parameters
    - vein wireframe interpretation
    - domain – data – estimation guidelines and North West data bias checks

2.5 **Personal inspections**

Authors David Gray and Michael Lawlor, each acting as a Qualified Person for this Technical Report, have each personally inspected the Kansanshi site at intervals from 2013 to 2014. The most recent QP site visits were in May 2015 and December 2014, respectively.

Robert Stone, also acting as a Qualified Person for this Technical Report, visited site in June and November 2014, and in February and April 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 relied on the commentary provided *verbatim* by Andrew Hester, FQM Environmental Manager, Africa.

ITEM 4 PROPERTY LOCATION, DESCRIPTION AND TENURE

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

4.1 Property location and area

The Kansanshi Operations are located in the north-western province of Zambia; approximately 15 km north of the town of Solwezi and 180 km northwest of the Copperbelt province town of Chingola (Figure 4-1).

Figure 4-1 Location of Kansanshi, North-western Province, Zambia (source: Titley, 2012)

The operations are located within Large Scale Mining Licence, LML 16, covering an area of 21,655 ha (enlarged to 24,865 ha in November 2014), within which there are surface rights to over 7,217 ha.

4.2 Tenure

FQM acquired its 80% interest in the Kansanshi Project in 2001 from Cyprus Amax Minerals Corporation. The remaining 20% is owned by a subsidiary of ZCCM. The site operating entity is Kansanshi Mining PLC (KMP).

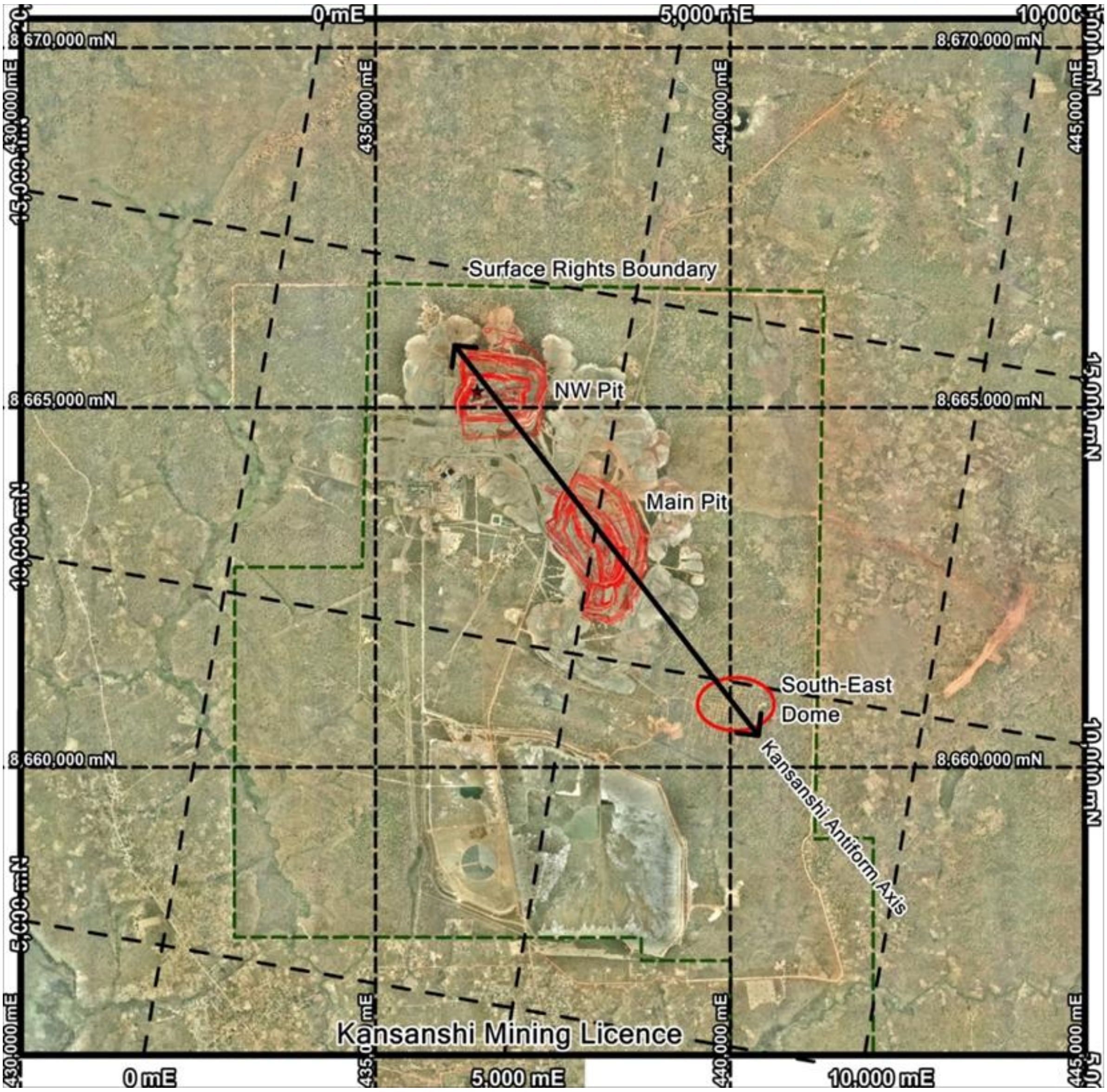
KMP is the owner of the LML 16 mining licence, which was issued on the 7th March 1997 and is valid until 7th March 2022 whereupon renewal can be applied for. The mining licence allows KMP to explore and mine copper, cobalt, gold, silver, tellurium, selenium and sulphur.

LML 16 was converted from its original shape under the 1995 Mines and Minerals Development Act to a compliant shape under the 2008 Mines and Minerals Development Act to license No: 7057-HQ-LML. The extents of LML16 are defined in Table 4-1 and shown in Figure 4-2.

Table 4-1 Extents of Kansanshi Mining Licence, LML16 (coordinates are in UTM)

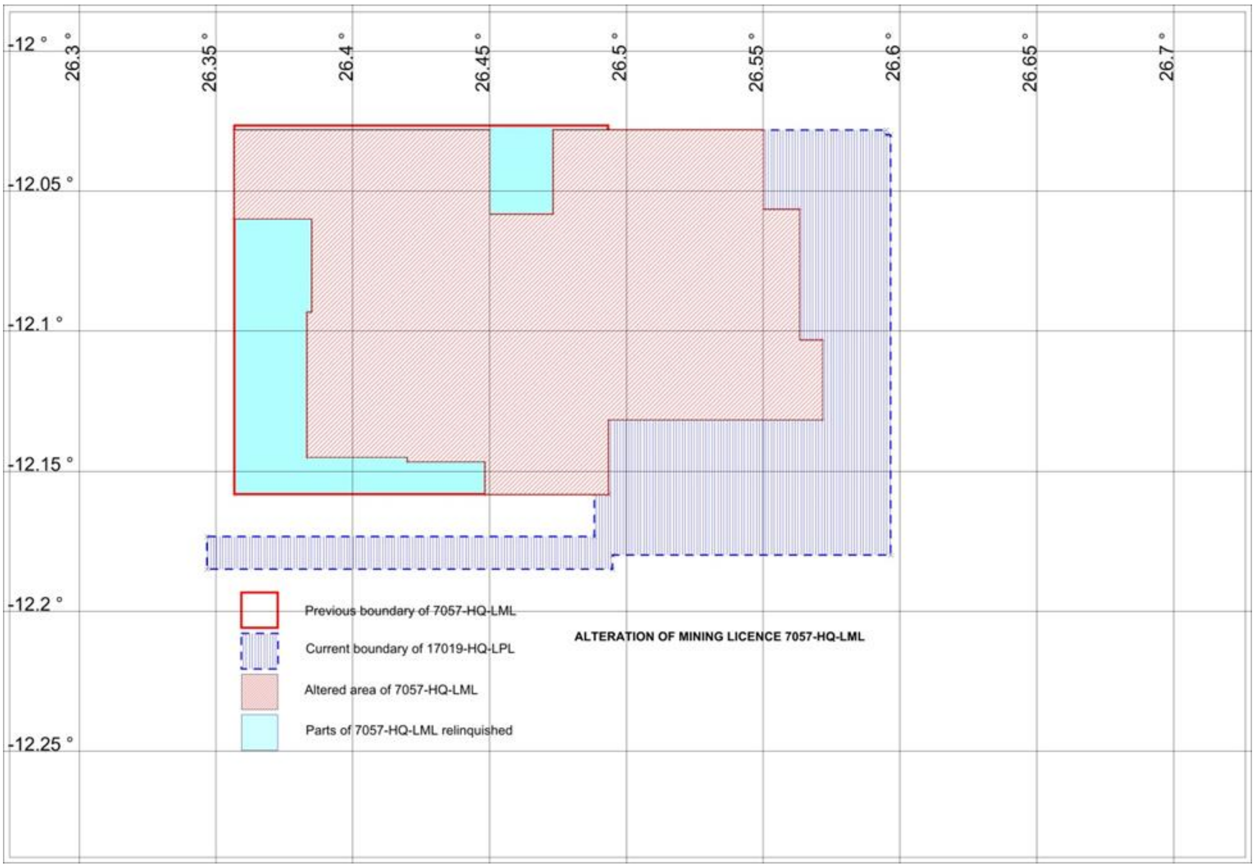
Side	Distance (m)	Direction Angles	Point	Northing	Easting
KMP1- KMP2	14,876.967	089° 52’ 48.65”	KMP1	8,670,394.289	430,001.622
KMP2- KMP3	14,558.848	179° 53’ 37.88”	KMP2	8,670,385.739	444,974.940
KMP3 - KMP4	14,869.690	269° 52’ 43.95”	KMP3	8,655,993.575	444,968.327
KMP4- KMP1	14,559.187	359° 51’ 54.8”	KMP4	8,655,999.556	429,985.218

Figure 4-2 Location of the Kansanshi Mining Licence and Surface Rights



On 3rd September 2014, KMP applied to the Ministry of Mines, Energy and Water Resources for the re-orientation of the Kansanshi mining lease number 7057-HQ-LML. This application was a forward-looking move aimed at increasing the size of the lease to allow for further infrastructure expansions as well as excluding areas compromised by encroachment. The application was approved on the 7th of November 2014 and the area of the lease is now 24,865 ha for the expanded licence area shown in Figure 4-3.

Figure 4-3 Expanded Kansanshi Mining Licence Area, November 2014



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 Kansanshi 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 decided to amend the royalty increase to 9% and retain the income tax and profits taxes at the 2014 levels.

The optimisation work forming the basis of the Mineral Reserves estimate was completed prior to April 2015, and a royalty rate of 6% was included amongst the optimisations inputs at that time.

Item 15.4.5 describes the impact on the adopted optimal pit shell due to royalties being increased to 9%.

KMP is the surface rights owner of four properties located within the LML16 mining license. These are Farm 724 (4,887 ha), Lot 18/M (2.2 ha), Lot 1514/M (2,288.9 ha), and Lot 936/M (38.5 ha). The farm is a one hundred year lease and the lots are all ninety-nine year leases.

In respect of LML 16, a licence fee is payable upon grant of the licence and under the current Regulations, amounts to K32,000. There is also an annual area charge payable on or before the anniversary of the grant of the licence. For a large-scale mining licence such as Kansanshi the annual area charge is calculated as follows:

- official area of licence = 248.64655 km^2
- area charge fee units per km^2 = 5,600
- fee unit value = K0.20
- annual area charge = K1,120/km^2
- annual area charge for Kansanshi = K1,120 x 248.64655 = K278,484.14

4.4 Environmental liabilities

The main environmental liabilities at Kansanshi 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 Kansanshi Mine closure plan is reviewed annually.

At the 31st December 2014, the unplanned asset retirement obligation (ARO) at Kansanshi was estimated to be $90.664 million. In accordance with National Legislation, Kansanshi contributes to an Environmental Protection Fund administered by the Zambian Mines Safety Department.

The Company is implementing a number of environmental standards in accordance with the ISO 14001 Environmental Management System. The standards provide a structured approach to environmental management including pollution prevention, legal compliance and continued environmental improvement.

No material environmental incident was reported at Kansanshi as at the end of May 2015 and the Company has not been subject to any penalties arising as a result of water pollution or contamination of land beyond the boundaries of its operation. To the Company's knowledge, KMP is not considered by any applicable environmental regulatory authority to be an imminent threat to the environment.

4.5 Potential access and exploitation risks

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

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

The following information is reproduced from the December 2012 Technical Report (Titley, CSA, 2012). There is no change in this information since reported in December 2012, other than an update by Michael Lawlor (QP) on the number of persons employed at the Kansanshi Operations and the sufficiency of surface rights. The information remains current and relevant.

5.1 Accessibility

The T5 highway provides the primary access to the Kansanshi Operations site. The highway is maintained by FQM on a regular basis and is in a reasonable condition. The highway is a tarred national road linked to the Copperbelt towns to the south-east and to the town of Mwinilungu to the west.

The closest airport is located in Solwezi, approximately 15 km from the mine site.

5.2 Climate

The region has distinct dry (April to October) and wet (November to March) seasons. Rainfall typically occurs as heavy thunderstorms producing between 10 and 40 mm of rainfall. The mean annual rainfall is around 1,400 mm. Mean temperatures range from between 5°C and 25°C in June and 14°C and 30°C in October. The climate has minimal impact on continuous mining and processing operations.

5.3 Physiography

The Kansanshi Operations lie on the Central African Plateau at an altitude of approximately 1,400 m (a.s.l.). The topography of the area is undulating with gentle slope gradients of between 1 and 2%. The watershed dividing the Congo and Zambezi River basins follows the DRC/Zambia border which lies approximately 18 km to the north. Surface drainage from the Kansanshi site thus flows predominantly south towards the Mutanda River, part of the Kafue River system, eventually draining into the Zambezi River. Direct surface runoff from the Kansanshi deposits flows into a small wetland or dambo approximately 2 km to the south of the mine site.

5.4 Vegetation

The natural vegetation type in the area is Miombo woodland, a semi-deciduous broadleaf vegetation type found from Angola to Burundi.

5.5 Local resources

The nearest major population centre is Solwezi, 15 km away. The estimated population is approximately 200,000 people, most of whom live in rural areas surrounding the town. Personnel can be and are recruited from this local community. Whilst the majority of local people are unskilled and require training, skilled artisans and professional people can be and are recruited from throughout northern Zambia.

As at the end of December 2014, the Kansanshi Operations employed over 2,900 persons directly and over 4,500 contractors.

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

5.6 Infrastructure

Prior to Project acquisition, the infrastructure in the Solwezi area was poor. Roads, the airport, hospitals and schools were in need of significant upgrades. As a result, the Company undertook a number of measures to improve infrastructure including the construction of a new power line into Solwezi, and the upgrading of the main road between Solwezi and the Project site, both completed in 2004. The main sealed road from Chingola to Solwezi was repaired and upgraded in 2002, whilst the existing airstrip at Solwezi was equipped with a tower and radio control.

Improved power supply to the Kansanshi site is supplied through the parastatal company Zambia Electricity Supply Corporation Ltd (ZESCO). Current KMP consumption of around 129 MVA is expected to increase to 230 MVA as a result of the site expansion projects. Hydroelectric generation from the Kariba South Bank Dam, the Kafue Gorge Dam and the Itezi Itezi Dam supplies power into the ZESCO grid. A single 330kV line runs from the Copperbelt to the KMP site from where a ZESCO substation transforms the power down to 33kV for mine usage.

5.7 Sufficiency of surface rights

The LML 16 mining licence boundaries were expanded in November 2014 (Item 4.2), sufficient to enable continued waste dumping to the east and north east of the Main and North West Pits. As could be appreciated from Figure 4-2, the licence area and the space for waste dumping had been previously constrained.

ITEM 6 **HISTORY**

The following information is reproduced in part from the December 2012 Technical Report (Titley, CSA, 2012). Michael Lawlor (QP) has provided updated information in respect of historical Mineral Resource and Mineral Reserve estimates, and events leading up to the current status of production.

6.1 Prior ownership

Modern day mining commenced at Kansanshi in 1969 when ZCCM developed an open pit mine to access high grade copper oxide ore. Intermittent mining and processing continued until 1986 when operations ceased due to prevailing economic conditions. In 1988, after a resumption of mining operations, ZCCM constructed a small sulphide flotation plant for the supply of concentrate to an offsite smelter. In 1998, ZCCM formally ceased operations at Kansanshi and initiated closure and reclamation activities.

Cyprus Amax Minerals Corporation (Cyprus Amax) subsequently entered into an agreement with ZCCM to acquire 80% of the rights to the Kansanshi Project, and these rights were in turn acquired by FQM in August 2001.

6.2 Exploration and development work

Cyprus Amax carried out two phases of diamond drilling between 1997 and 1999, totalling 83,441 m from 379 diamond drilled (DD) holes. In addition to this, a programme of 52 reverse circulation (RC) drill holes was undertaken for exploration, condemnation and water monitoring purposes.

Between 2003 and 2007, FQM undertook reverse circulation and diamond drilling in-fill programmes. Since 2010 there has been an ongoing intensive diamond drilling programme at Kansanshi, including drilling in the South East Dome region.

6.3 Previous Mineral Resource estimates

On behalf of the issuer, CSA produced the previous Mineral Resource estimate detailed in the 2012 Technical Report (Titley, CSA, 2012) which complies with the reporting requirements of the Canadian National Instrument 43-101 guidelines. At the time, it is the opinion of the QP, David Gray, that this estimate was both reliable and relevant. The 2012 estimate used the data available at the time to estimate copper, acid soluble copper and gold grades using ordinary kriging into an empty geology block model having vein and strata domains. The Mineral Resource estimate included the South East Dome deposit and the statement is reproduced in Table 6-1.

The 2012 Mineral Resource estimate is, however, no longer current due to the continued open pit mining exposure and diamond and RC grade control drilling coverage of the available mineralisation. The additional data gained since 2012 and the improved geology understanding in the context of mineralisation, requires updating, as per this Technical Report, in order to better reflect a more accurate and representative Mineral Resource estimate.

Table 6-1 2012 Mineral Resource estimate

KANSANSHI and SOUTH EAST DOME MINERAL RESOURCE ESTIMATE as at 30th November, 2012 at >= 0.3 Cu% cut-off						
Area	**Resource Classification**	**Million Tonnes**	**Cu Grade %**	**ASCu Grade %**	**Au Grade ppm**	**Cu metal kt**
KANSANSHI	Measured	88.9	1.10	0.48	0.17	974
Main and	Indicated	601.1	0.83	0.19	0.14	4,967
NW Pits	**Total M+I**	**690.0**	**0.86**	**0.23**	**0.15**	**5,942**
	Inferred	344.4	0.70	0.04	0.11	2,401
SE DOME	Measured	-	-	-		-
	Indicated	54.2	0.90	0.04	0.15	486
	Total M+I	**54.2**	**0.90**	**0.04**	**0.15**	**486**
	Inferred	20.8	0.91	0.03	0.16	188
TOTAL	Measured	88.9	1.10	0.48	0.17	974
	Indicated	655.4	0.83	0.18	0.14	5,453
	Total M+I	**744.3**	**0.86**	**0.21**	**0.15**	**6,427**
	Inferred	365.2	0.71	0.04	0.12	2,589

6.4 Previous Mineral Reserve estimates

DumpSolver Pty Ltd (DumpSolver) produced the previous Mineral Reserve estimate for Kansanshi (Journet and Cameron, DumpSolver, 2012). The open-pit mining inventory statement is reproduced in Table 6-2, whilst the accompanying stockpile inventory statement is listed in Table 6-3. In the opinion of QP Michael Lawlor, this estimate was relevant and reliable at the time of reporting. It 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, detailed pit designs and life of mine production schedules.

Since 2012, the metallurgy of feed types has been redefined as outlined in Items 1.5 and 13.1. The previously reported “leach ore” and “mixed ore” are now referred to respectively, as “mixed leach ore” and “mixed float ore”, or collectively as float/leach feed. The sulphide ore definition remains the same and is also referred to as float feed.

The 2012 Mineral Reserve estimate (Journet and Cameron, DumpSolver, 2012) is now replaced by the estimate reported and tabulated under Item 15.

Table 6-2 Kansanshi Operations Mineral Reserve statement, in-pit inventory at November 2012

MINERAL RESERVE AT 30th NOVEMBER 2012 (AT $3.00/lb Cu and $1200/ounce Au)							
Leach Ore							
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**	**TCu metal (kt)**	**Au metal (t)**
MAIN & NW	Proved	32.7	1.56	1.01	0.18	511	6
	Probable	74.5	1.24	0.76	0.15	921	11
	Total P+P	**107.2**	**1.34**	**0.84**	**0.16**	**1,433**	**17**
SE DOME	Proved	-	-	-	-	-	-
	Probable	0.5	0.94	0.43	0.07	4	0
	Total P+P	**0.5**	**0.94**	**0.43**	**0.07**	**4**	**0**
TOTAL	Proved	32.7	1.56	1.01	0.18	511	6
	Probable	75.0	1.23	0.76	0.15	926	11
	Total P+P	**107.7**	**1.33**	**0.84**	**0.16**	**1,437**	**17**
Mixed Ore							
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**	**TCu metal (kt)**	**Au metal (t)**
MAIN & NW	Proved	22.1	0.80		0.15	177	3
	Probable	113.3	0.69		0.13	786	14
	Total P+P	**135.4**	**0.71**		**0.13**	**963**	**18**
SE DOME	Proved	-	-		-	-	-
	Probable	9.7	1.30		0.22	126	2
	Total P+P	**9.7**	**1.30**		**0.22**	**126**	**2**
TOTAL	Proved	22.1	0.80		0.15	177	3
	Probable	123.1	0.74		0.13	912	17
	Total P+P	**145.2**	**0.75**		**0.14**	**1,090**	**20**
Sulphide Ore							
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**	**TCu metal (kt)**	**Au metal (t)**
MAIN & NW	Proved	32.3	0.65		0.14	209	4
	Probable	435.7	0.65		0.13	2,815	56
	Total P+P	**468.0**	**0.65**		**0.13**	**3,024**	**61**
SE DOME	Proved	-	-		-	-	-
	Probable	36.8	0.69		0.12	255	4
	Total P+P	**36.8**	**0.69**		**0.12**	**255**	**4**
TOTAL	Proved	32.3	0.65		0.14	209	4
	Probable	472.5	0.65		0.13	3,071	61
	Total P+P	**504.8**	**0.65**		**0.13**	**3,279**	**65**
Total Ore							
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**	**TCu metal (kt)**	**Au metal (t)**
MAIN & NW	Proved	87.1	1.03		0.16	898	14
	Probable	623.5	0.73		0.13	4,522	82
	Total P+P	**710.7**	**0.76**		**0.13**	**5,420**	**96**
SE DOME	Proved	-	-		-	-	-
	Probable	47.0	0.82		0.14	386	7
	Total P+P	**47.0**	**0.82**		**0.14**	**386**	**7**
TOTAL	Proved	87.1	1.03		0.16	898	14
	Probable	670.5	0.73		0.13	4,909	88
	Total P+P	**757.7**	**0.77**		**0.13**	**5,806**	**102**

Table 6-3 Kansanshi Operations Mineral Reserve statement, stockpile inventory at November 2012

MINERAL RESERVE AT 30th NOVEMBER 2012 (AT $3.00/lb Cu and $1200/ounce Au)							
Leach Ore							
S/Piles	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**	**TCu metal (kt)**	**Au metal (t)**
TOTAL	Proved	10.2	1.06	0.68	0.16	108	2
	Probable	-	-	-	-	-	-
	Total P+P	**10.2**	**1.06**	**0.68**	**0.16**	**108**	**2**
Mixed Ore							
S/Piles	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**	**TCu metal (kt)**	**Au metal (t)**
TOTAL	Proved	12.0	0.82		0.14	98	2
	Probable	-	-		-	-	-
	Total P+P	**12.0**	**0.82**		**0.14**	**98**	**2**
Sulphide Ore							
S/Piles	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**	**TCu metal (kt)**	**Au metal (t)**
TOTAL	Proved	9.2	0.58		0.13	53	1
	Probable	-	-		-	-	-
	Total P+P	**9.2**	**0.58**		**0.13**	**53**	**1**
Total Ore							
S/Piles	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**	**TCu metal (kt)**	**Au metal (t)**
TOTAL	Proved	31.4	0.83		0.14	260	4
	Probable	-	-		-	-	-
	Total P+P	**31.4**	**0.83**		**0.14**	**260**	**4**

6.5 Production history

FQM production commenced in 2004 following the construction of a 4 Mtpa oxide circuit and a 2 Mtpa sulphide circuit. Capital additions increased the sulphide circuit to 4 Mtpa, and by 2006 a second expansion had further increased sulphide production to 8 Mtpa. Another independent sulphide circuit was completed in early 2008 which allowed sulphide production to increase again to 12 Mtpa.

Further capital works during 2009 allowed for the treatment of transitional ore (mixed ore) through the original sulphide circuit, and following treatment optimisations each process stream could eventually cater for 5 Mtpa oxide, 5 Mtpa mixed and 12 Mtpa sulphide feed. Further capital works commenced in early 2010 to increase the sulphide processing capacity to 14 Mtpa, thereby providing an overall plant capacity of around 25 Mtpa (the S3 expansion project).

From 2004 to May 2015 open pit mining at Kansanshi has produced 2.2M tonnes of copper to date (Table 6-4).

Table 6-4 KMP Copper production from Kansanshi Operations, to date

Year	Copper in Concentrates				Cathode	Total Cu Production
	Sulphide	Mixed	Oxide	Total		
	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes	Tonnes
Prior 2005						80,000
2005	32,295	0	2,036	34,332	43,425	77,757
2006	51,851	0	7,362	59,213	67,651	126,863
2007	57,938	0	12,638	67,334	96,497	163,831
2008	103,450	0	9,518	101,783	113,538	215,321
2009	107,625	32,171	13,140	150,370	94,609	244,979
2010	74,009	47,472	22,961	142,550	88,574	231,125
2011	57,115	55,579	22,644	120,822	109,474	230,296
2012	83,082	64,615	19,030	156,675	104,677	261,352
2013	81,968	65,132	26,480	161,270	113,467	274,737
2014	68,233	75,624	18,599	140,218	122,069	262,287
2015 Ytd	20,750	40,654	5,493	58,430	33,993	92,423
Total	738,318	381,248	159,901	1,192,996	987,974	2,180,970

* Production prior to 2005 is an estimated figure

Note that the totals may not add up due to concentrate sent to the High Pressure Leach circuit, which is recovered in the cathode circuit.

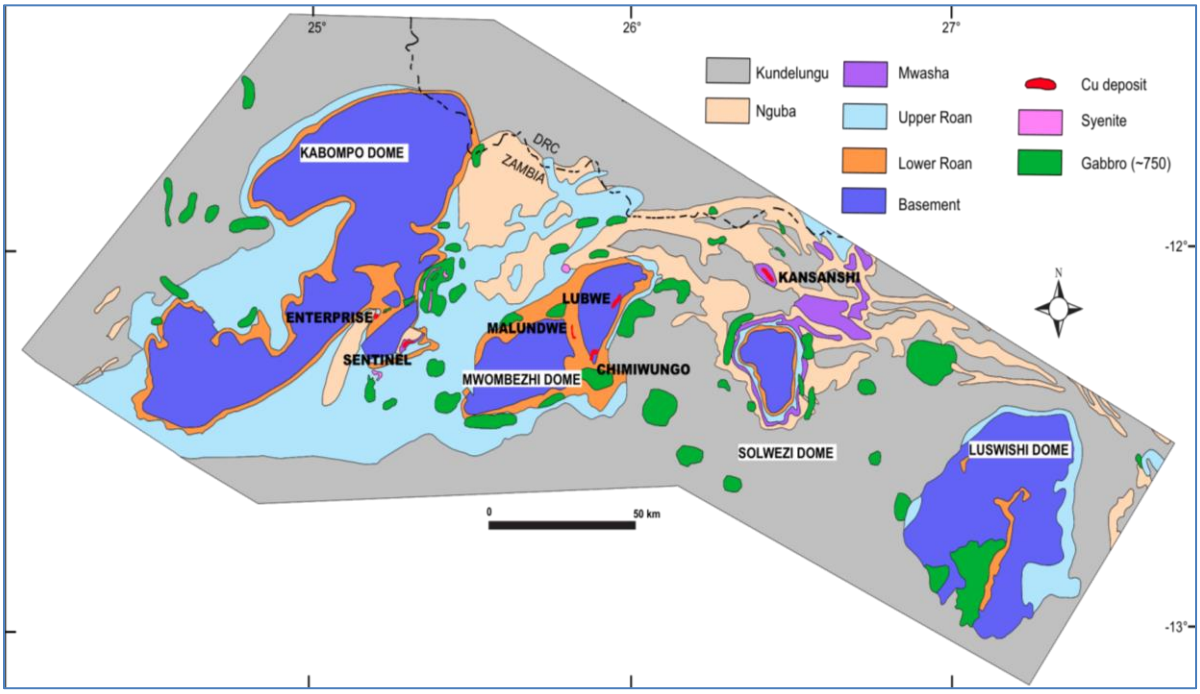
ITEM 7 GEOLOGICAL SETTING AND MINERALISATION

The following information is reproduced from the December 2012 Technical Report (Titley, CSA, 2012). There is no change in this information since reported in December 2012, and it remains current and relevant.

7.1 Regional, local and property geology

The Kansanshi deposit is located in the "Domes Region" of the North-Western Province of Zambia and is hosted within the Katangan Supergroup sediments of the Central African Copper belt (CAC). The deposit is believed to be hosted within the metasediments of the Nguba (previously referred to as the Lower Kundulungu), Grand Conglomerate, and Mwashya Groups. Regional stratigraphy is tabled in Table 7-1 and the regional surface geology plan is presented in Figure 7-1[1].

Figure 7-1 Simplified Regional Geology of the "Domes Region" of North-western Province, Zambia



A major basement feature known as the Solwezi Dome, comprised of granites, migmatites and gneisses, is located approximately 12 km south of Kansanshi (Figure 7-1). A series of metamorphosed schists, quartzites and conglomerates are exposed around the Solwezi Dome Margins and are assumed to be the chronological equivalents of the Lower Roan Group, a characteristic unit within the Zambian Copperbelt (Cyprus Amax, 2000). Banded ironstones, quartzites and phyllites exposed along the northern edge of the Dome are thought to represent the Mwashia Group, Arthurs, 1974 (Figure 7-1).

[1] Coordinates are in WGS84, LatLon, Image is compiled from Tembo 1994, Armstrong et. al. 1999, Torrealday, 2000, Broughton, pers. comm., and Key et. al. 2002.

Locally, calcareous-dolomitic, quartz-biotite-hornblende-garnet schist can be found between the rocks surrounding the Solwezi Dome and the Kansanshi Mine area. Drilling within the Kansanshi Mine area has confirmed that these schists overlie the Kansanshi mine Sequence. The Kansanshi mine sequence is thought to be recumbently folded Katangan-aged metasediments with a pronounced NW-SE trending antiformal structure, known as the Kansanshi Antiform.

Along the strike of the Kansanshi Antiform re-folding has occurred creating doubly-plunging, domed structures along its crest. It is within these domed structures that the three major ore bodies (NW Pit, Main Pit and SE Dome) of the Kansanshi Mining License are located. A summary of the Kansanshi Mine Stratigraphic Sequence is outlined in Table 7-2.

For a comprehensive description of the geological, structural, and tectonic setting of the Kansanshi Deposit please refer to Cyprus Amax PFS (2000), Beeson et al. (2008).

Table 7-1 Katangan Stratigraphic Column for the Zambian Copperbelt (Cyprus Amax, PFS, June 2000).

SYSTEM	SERIES	COPPERBELT GROUP/ FORMATION		SOLWEZI FORMATION	SOLWEZI FORMATION (after Arthurs, 1974)	SOLWEZI LITHOLOGY
KATANGA SUPER GROUP	LOWER KUNDELUNGU	Lower Kundelungu Shale Fm	UNCERTAIN CORRELATION	Solwezi Biotite-Quartz Fm	Solwezi Biotite-Quartz Fm	(calc) biotite-hornblende-quartz- musc-schists, phyllite
		Kakontwe Limestone		Upper Dolomite Fm	Chafugoma Marble Fm	Dolomite, minor phyllite
				Upper Pebble Schist Fm	Pelitic Fm	Calcareous pebble schist, knotted schist, bio-musc schist; scapolite
				Kansanshi Mine Fm	Pelitic Fm	Marble, calc biotite schist, locally carbonaceous phyllite, knotted schist; scapolite
		Grand Conglomerate/ Tillite Fm		Lower Pebble Schist Fm	Paraconglomerate Fm	Calcareous pebble schist, knotted schist, bio-musc schist; scapolite
	(COPPER BELT) MINE SERIES	Mwashia Group (argillaceous)		Mwashia?	Chafugoma Marble Fm Lower Unit ("Tectonic melange")	Marbles, calc-biotite schists, scapolite Calc-bio schists, marble, (local ironstone), scapolite
		Upper Roan Group (dolomitic)		Upper Roan?	Upper Roan?	Dolomite-muscovite schists
		Lower Roan Group (arenaceous)		Lower Roan?	Lower Roan?	Quartzite, quartz-mica schists

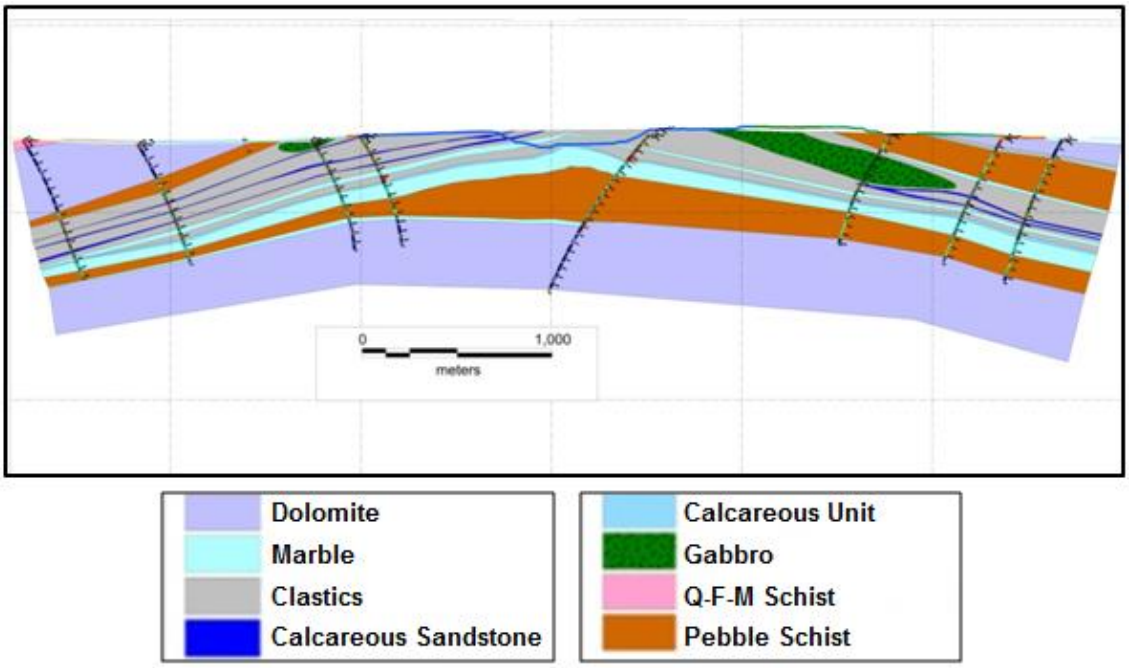
Locally, calcareous-dolomitic, quartz-biotite-hornblende-garnet schist can be found between the rocks surrounding the Solwezi Dome and the Kansanshi Mine Area. Drilling within the Kansanshi Mine area has confirmed that these schists overlie the Kansanshi Mine Sequence. The Kansanshi Mine Sequence is thought to be recumbently folded Katangan-aged metasediments with a pronounced NW-SE trending antiformal structure, known as the Kansanshi Antiform (Figure 7-2).

Along the strike of the Kansanshi Antiform form re-folding has occurred creating doubly-plunging, domed structures along its crest. It is within these domed structures that the three major ore bodies (NW Pit, Main Pit and SE Dome) of the Kansanshi Mining License are located. A summary of the Kansanshi Mine Sequence is outlined in Table 7-2. For a comprehensive description of the geological, structural, and tectonic setting of the Kansanshi Deposit please refer to Cyprus Amax PFS (2000), Beeson et al. (2008).

Table 7-2 Kansanshi Mine (Stratigraphy) Sequence, *(after Cyprus Amax report, 2000)*

Sequence	Code	Unit	Lithology
KANSANSHI MINE SEQUENCE	QFMS	Quartz-Feldspar-Mica Schist	Quartz-Feldspar-Mica-Schist
	UD	Upper Dolomite	Dolomite
	UPS	Upper Pebble Schist	Pebble schist
	TMM	Topmost Marble	Marble
	UMC	UPPER MIXED CLASTICS	Knotted Schist Carbonaceous Phyllite Knotted Schist \ Carbonaceous Phllite Calcareous Sandstone Carbonaceous Phyllite Calcareous Sandstone Knotted Schist Carbonaceous Phyllite
	UM	UPPER MARBLE	Marble \ Residual Marble Phyllite \ Carbonaceous Phyllite
	MMC	MIDDLE MIXED CLASTICS	Phyllite \ Biotite Schist Knotted Schist Carbonaceous Phyllite Calcareous Phyllite \ Sandstone Knotted Schist
	LCS	LOWER CALCREOUS SEQUENCE	Calcareous Biotite Schists Knotted Schist \ Phyllites
	LM	LOWER MARBLE	Marble Phyllite \ Carbonaceous Phyllite \ KS Marble
	LPS	LOWER PEBBLE SCHIST	Pebble schist
	LD	LOWER DOLOMITE	Marble \ Carbonaceous Phyllite Dolomite
	MCS	Micaceous Clacareous Schist	Micaceous Calcareous Schist

Figure 7-2 Regional cross section showing the Kansanshi Antiform Axis



7.2 Alteration types

Alteration at Kansanshi is typically associated with halos around mineralised veins. Alteration types at Kansanshi are described in terms of type and intensity and are strongly controlled by lithological units. A detailed report on the alteration types of the Kansanshi can be found in the Cyprus Amax PFS (Cyprus Amax, 2000).

A summary of the alteration types are presented in Table 7-3.

Table 7-3 Alteration types seen at Kansanshi

Code	Type	Description
CA	Calcite	Introduced metasomatic or hydrothermal calcite, commonly found in the LCS and in alteration halos within Marble units where this alteration type is characterized by destruction of dark minerals and fine-grained carbon producing a whitened Marble.
CH	Chlorite	Introduced metasomatic or hydrothermal chlorite. Not to be confused with metamorphic chloritization.
DD	De-dolomitization	Incomplete de-dolomitization of the feldspathic dolomite rocks frequently producing increased porosity. Difficult to distinguish from dissolution via weathering. Commonly found in the LCS.
DO	Dolomitization	Introduced hydrothermal dolomite, ranges from fine to coarse grained and granoblastic.

GM	Green Mica	Conspicuous bright to emerald green crystals of (chromian?) muscovite. Usually associated with PG and CA alteration halos.
PF	Potassium Feldspar	Pinkish alteration locally developed around veins, particularly adjacent to marble contacts. Not to be confused with the pink/orange staining produced by the weathering of iron oxides. Usually associated with PG alteration halos.
PG	Plagioclase Feldspar	Pervasive replacement of dolomite, graphite and mica by albite with associated introduction of rutile, ankeriterhombs and bright green (chrome-bearing) muscovite. Produces a bleached rock. Weathers to white kaolinitic saprolite.
SC	Scapolite	Millimetre to centimetre sized subhedral to euhedral prismatic scapolite crystals are commonly developed in muddy carbonates and calcareous schists at marble – schist contacts. Colour varies from dark to pale grey, to pale green or bleached grey where the scapolite is affected by later dolomitic or albitic alteration. Hand specimen and petrographic study shows it overprints metamorphic biotite, sericite, plagioclase, quartz.
SE	Sericite	Hydrothermal yellowish sericite (different from the green muscovite), occurs along vein selvedges, as select bands or disseminations. Excludes metamorphic sericite.
SL	Silica	Any alteration logged as “silicic” or “silicified”.

7.3 Structure

Mineralisation at Kansanshi is strongly influenced by structural deformation including dome geometry and proximity to the Antiform axis. The Main Pit, North West Pit and South East Dome Pit are located on medium to large scale parasitic domes along the crest of the Kansanshi Antiform (Figure 7-3).

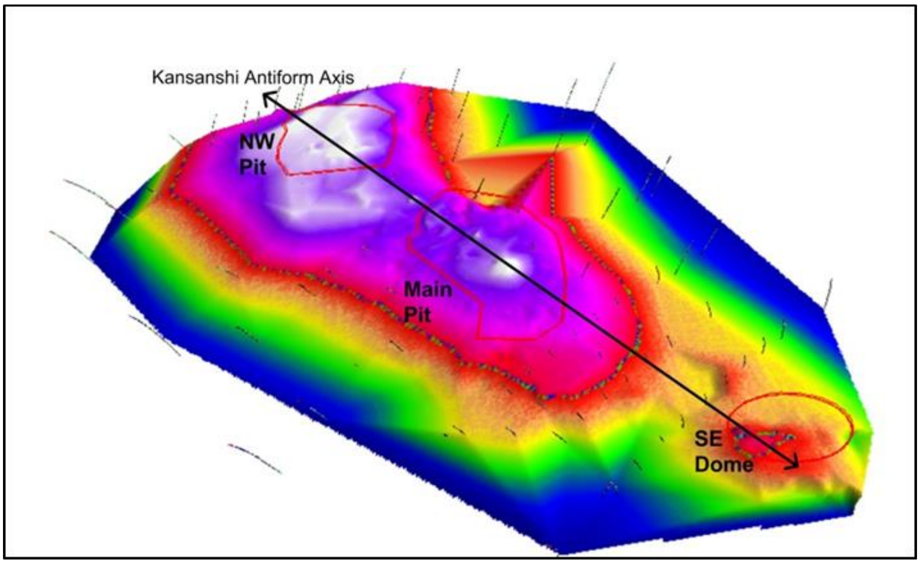
The Main Pit has two structural zones, known as the 4800E and 5400E zones, corresponding to the rough location on the local grid. The 4800E and 5400E structures are north-south zones of faulting, veins, and breccia. Both zones are deeply weathered with considerable oxide mineralisation associated with the veining and brecciation.

Mineralisation at Main Pit is present below the apex of the dome in all stratigraphic units, including the Lower Pebble Schist and the Lower Dolomite unit. Current drilling away from the dome in the Main Pit shows that mineralisation stops at the bottom of the Middle Mixed Clastics (MMC) unit. There are some mineralised veins in the Lower Marble and Lower Pebble Schist off the dome; however, the majority of the veins are barren calcite-quartz at depth. In North West Pit there is limited to no mineralisation below the MMC.

Variations in the vein orientation can be seen between Main Pit, North West Pit and South East Dome. In summary:

- The Main deposit exhibits a steeply dipping, generally north-south trending vein set and a flat lying set found sub parallel to lithology in the Lower Marble. Within the Main deposit, small-scale radial veins surrounding the apex of the dome structure are also seen.
- The North West deposit has four steeply dipping vein sets with fracture mesh geometries.
- The South East Dome deposit has a steeply dipping NW-SE trending vein set.

Figure 7-3 Contouring of the top of the Upper Marble, showing the North West, Main and South East Dome deposits along the antiform axis



7.4 Mineralisation

7.4.1 Introduction

Mineralisation at Kansanshi is associated with veins, select lithologies and occasional fault breccia zones. Styles of mineralisation are mainly oxide, secondary sulphide and primary sulphide with distinct zones of mixed oxide and sulphides. Sulphide mineralisation is dominated by chalcopyrite with minor bornite and some gangue pyrite, the latter being more prevalent in NW Pit. The dominant oxide mineral assemblages are malachite and chrysocolla. Secondary sulphides are dominantly in the form of chalcocite. Mineralisation tends to favour the clastic than carbonate lithologies, with the Middle Mixed Clastics unit hosting the majority of vein and strataform mineralisation. Disseminated mineralisation is associated with varying degrees of alteration adjacent to the veins.

7.4.2 Vein mineralisation

Veins at Kansanshi are of a quartz/carbonate composition with accessory copper bearing minerals (Figure 7-4). Veins are mostly sub vertical with widths ranging from centimetres to meters. Chalcopyrite is the dominant sulphide copper bearing mineral while malachite and chrysocolla are the dominant oxides (Figure 7-5).

Veinlets less than 100 mm are logged as stringer veins. Stringer vein mineralisation occurs mostly within the sulphide zone and dominates mineralisation within the Lower Pebble Schist of Main Pit.

Minor vein constituents, such as anhydrite and magnetite, as well as trace uraninite, appear to increase with depth. Molybdenite is a common accessory mineral present in both Main and North West Pit calcite veins.

Figure 7-4 Vein hosted copper mineralisation in the Sulphide Zone at Kansanshi

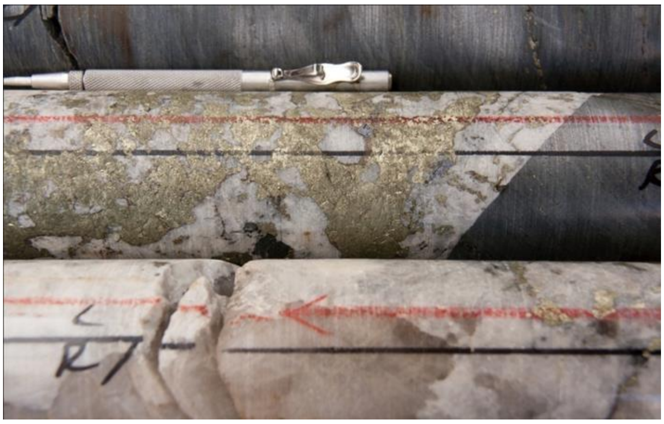
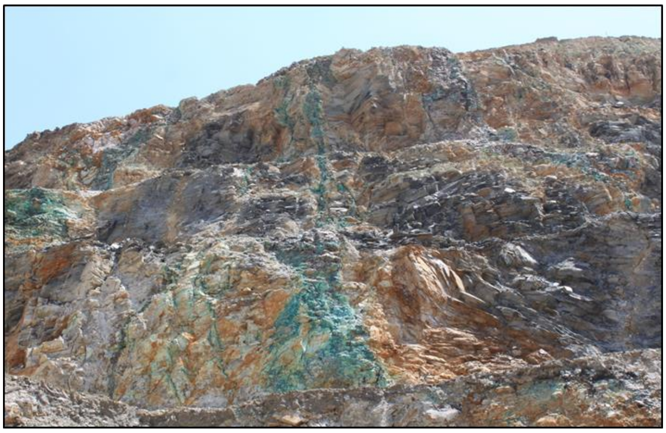


Figure 7-5 Malachite, chalcocite assemblages within oxidised calcitic veins



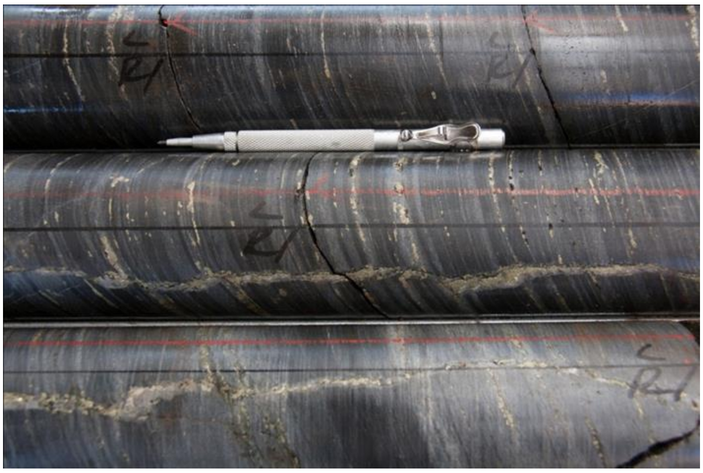
Mineralised veins often have adjacent zones of varying albite alteration, bleaching the host stratigraphy. These haloes vary from millimetres to several meters and appear to be influenced by adjacent lithology. The alteration haloes are typically large and diffuse within carbonaceous units and more constrained within the garnetiferous and knotted schists. Disseminated mineralisation is often associated with these haloes and is commonly present in the phyllites, carbonaceous phyllites and marble units (Figure 7-6).

Figure 7-6 Disseminated mineralisation seen in Marble within the Sulphide Zone

7.4.3 Strata bound mineralisation

Strata-bound mineralisation usually occurs within the sulphide zone. Bedding-parallel disseminated chalcopyrite is more dominantly present within phyllites and carbonaceous phyllites (Figure 7-7). Organic carbon within the phyllites acted as a redox boundary, encouraging the precipitation of sulphides. Sulphide species are typically chalcopyrite, bornite, pyrite and pyrrhotite.

Figure 7-7 Strata-bound and stringer vein mineralisation in carbonaceous phyllite



7.4.4 Breccia

Stock-work-breccia style mineralisation occurs where veining intensity increases (Hanssen et al., 2010). This is seen predominantly in the Main Pit dome region and is likely to also be associated with zones of increased structural deformation. In this area all stratigraphic units are host to mineralisation. There is also considerable breccia style mineralisation associated with the structural zones of the 4800E and 5400E.

7.5 Material types

7.5.1 Refractory

Refractory mineralisation at Kansanshi refers to elemental copper contained in a clay matrix that is not recoverable by normal processing methods. Elemental copper is contained within a matrix of

Smectite clays and iron oxide and hydroxide minerals. Refractory ore is found predominantly in the northern area of Main Pit, in the saprolite zone of the Upper Mixed Clastics.

7.6 Residual

Residual mineralisation is formed at Kansanshi by the dissolution of marble units during weathering with enrichment occurring by typical supergene processes.

7.6.1 Oxide mineralisation

The oxide zone of mineralisation at Kansanshi is comprised of malachite, tenorite, chrysocolla and minor azurite. Oxide mineralisation is observed in veins, alteration haloes and surrounding lithologies. Margins within the oxide zone are less defined due to the mobilisation and dispersion of the copper minerals during weathering. This complexity is exacerbated by the frequency of weathering along and down vertical vein sets; as a result, oxidized mineralisation can occur at depth and below the average depth of weathering. Gangue minerals are predominantly smectite clays, calcite, marble, iron oxides and hydroxides.

7.6.2 Mixed mineralisation

Mixed mineralisation occurs at the transition from sulphide to oxide mineralisation. The material is of mixed mineralogy containing primary copper sulphides, secondary copper sulphides and copper oxides. The ratio of copper to acid soluble copper and the overall acid soluble content are key in determining which processing circuit will yield optimal copper recovery.

7.6.3 Sulphide mineralisation

The sulphide copper at Kansanshi is predominately chalcopyrite with minor bornite. Gangue minerals are predominantly pyrite with minor pyrrhotite.

ITEM 8 DEPOSIT TYPE

The following information is reproduced from the December 2012 Technical Report (Titley, CSA, 2012). There is no change in this information since reported in December 2012; it remains current and relevant.

8.1 Mineral deposit types

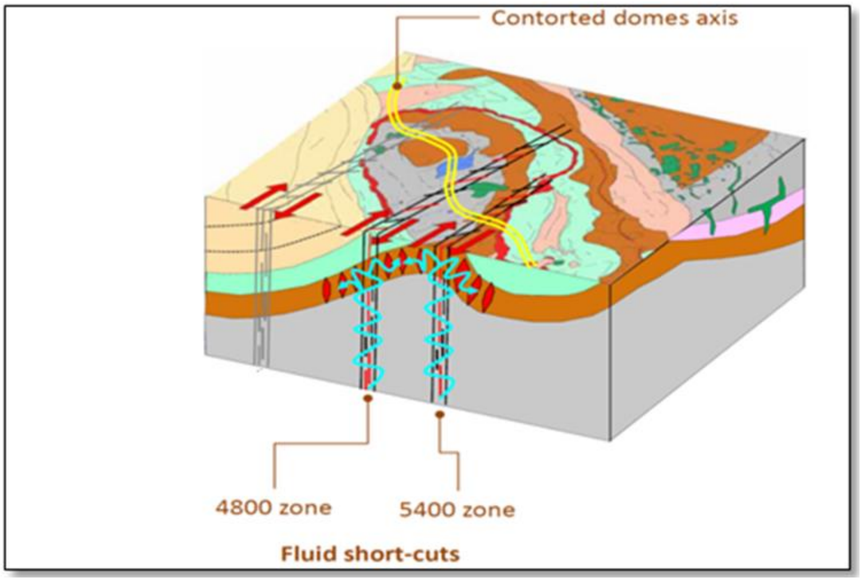
Kansanshi mineralisation occurs as hydrothermal, structurally controlled, vein and strata hosted copper deposits. Copper mineralisation developed in the Cambrian period as two discrete mineralisation pulses at ca. 512 Ma and ca. 502 Ma (Torrealday et al, 2000).

The origin of the oxidised copper enriched hydrothermal fluids is understood to have a magmatic origin which subsequently migrated upwards acquiring a significant mineral content through leaching and scavenging of elemental copper and gold from basement and/or overlying sedimentary rocks (Figure 8-1). Precipitation of copper mineralisation across the Kansanshi deposits was dependent upon the dome shaped trap structures and reducing conditions within the host lithologies.

The 4800E and 5400E structural zones are good examples of possible fluid pathways and traps for sulphides containing copper mineralisation. Vein type deposits are characterised by multiple mineralisation and remobilisation stages.

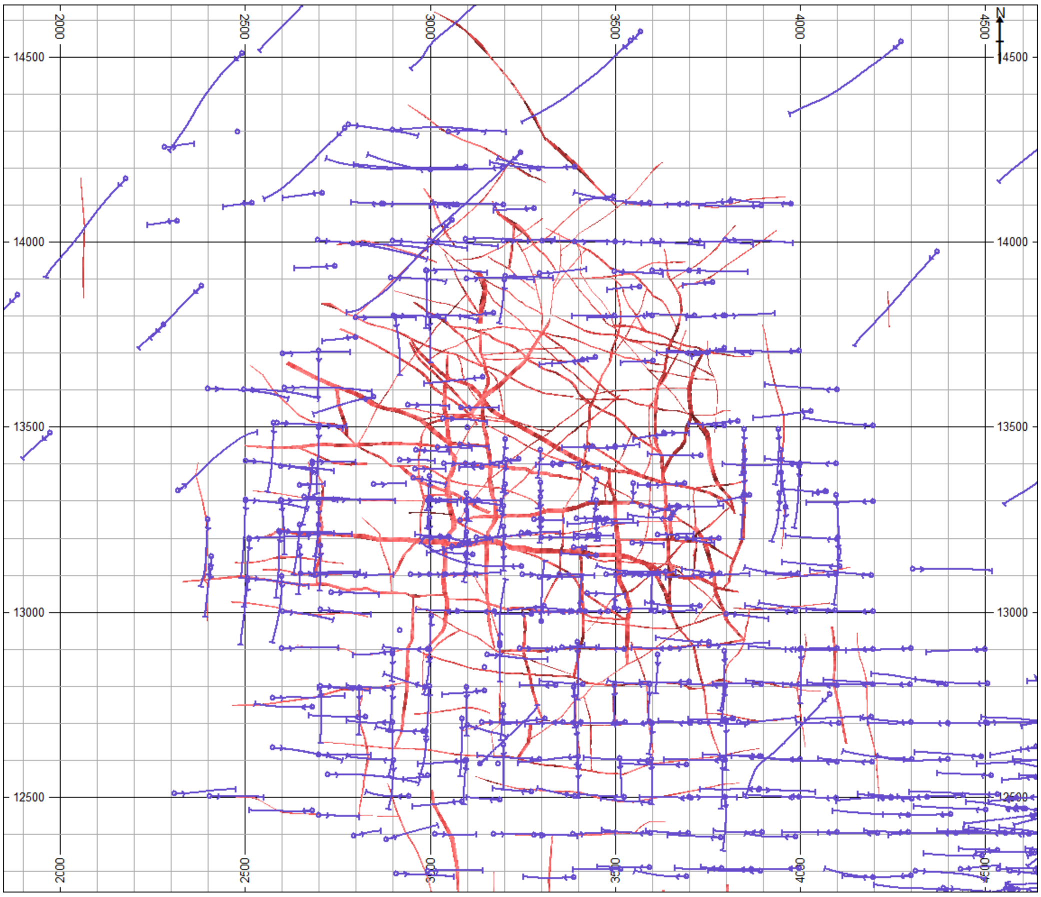
Sulphide (and associated metal) deposition occurred at redox interfaces where oxidised copper enriched fluids intercepted reducing carbonaceous sediments such as carbonaceous phyllites. Interaction between graphitic organic matter and hydrothermal fluid is likely to have controlled the oxidation state of the ore fluid and could have been the controlling factor leading to ore precipitation at Kansanshi (Kribek et al.,2005).

Figure 8-1 Genetic model showing possible pathways for hydrothermal fluid migration



These geological deposit concepts have guided exploration work and modelling approach. Specifically, drilling (Figure 8-2) has focussed on multi-directional holes drilled at approximately 60 to 70 degrees in order to maximise the vein intersection angles as well as the multiple vein orientations. Multiple direction and angled holes similarly optimise the angle of intersection with the antiformal shape of the lithologies.

Figure 8-2 A plan view of diamond drilling across the Kansanshi North West deposit, highlighting different drill orientations for optimal vein intersection. Red lines represent the modelled vein volumes.



At South East Dome, initial drilling focused on defining the shape and extent of the dome structure. Holes were drilled to a 100 m grid, with holes oriented both N-S and E-W. Sectional interpretations of the drilled logging data were used to guide the extents and infill drilling down to a 50 m grid. Grid spacing tended to be closest in the better mineralised dome apex areas and increased outwards to the extents of mineralisation.

Modelling approach has focussed on the core elements of the deposit style of mineralisation with a focus on delineating and modelling vein volumes and the key strata hosting mineralisation. These volume entities constitute key domains for mineral resource estimation in that they have continuous geology and grade behaviour.

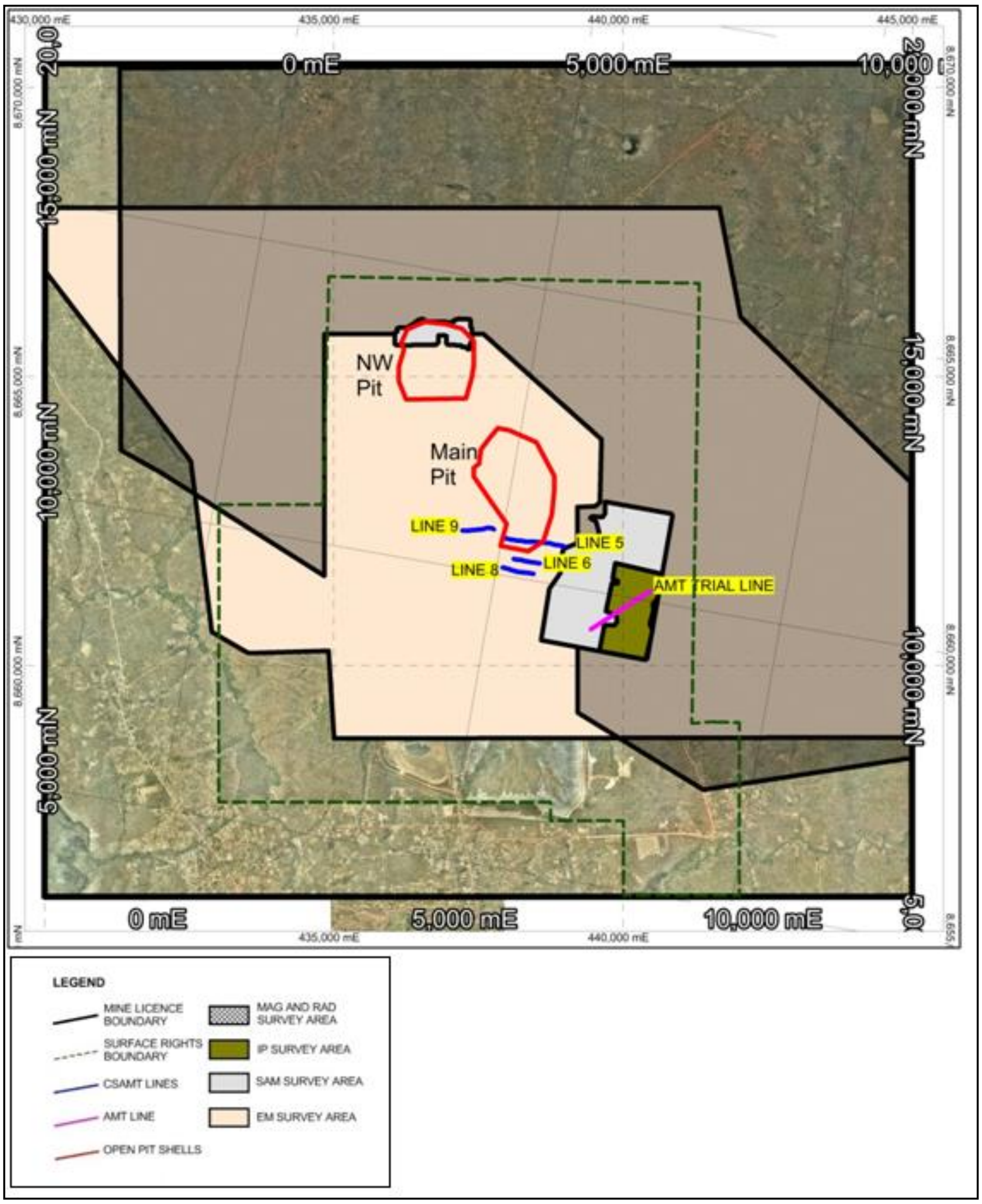
ITEM 9 **EXPLORATION**

The following information is reproduced from the December 2012 Technical Report (Titley, CSA, 2012). There is no change in this information since reported in December 2012; it remains current and relevant. Apart from drilling (covered in Item 10) there has been no exploration sampling completed across Kansanshi.

9.1 **Introduction**

Beginning in 2007, FQM commissioned a number of geophysical surveys to explore for additional copper mineralisation as well as to help define the limits of the Kansanshi mineralisation system. The geophysical methods used over the licence area and the areas that were covered are shown in Figure 9-1.

Figure 9-1 Location map for areas covered by geophysical surveys



9.2 Airborne Electromagnetic survey (EM)

In 2007, FQM contracted FUGRO Airborne Surveys (Fugro) from Johannesburg for a time-domain airborne electromagnetic survey (EM). The survey was undertaken over 107 km^2 of the Kansanshi Mining licence as shown in Figure 9-1. This survey employed a 75 Hz Tempest system with 100 m east–west line spacing.

The data processing was conducted by Darren Burrows from Fugro to produce conductivity depth images (CDIs) and channel levels. However, the quality of the output did not meet specifications due to the fact that Fugro's flight height varied from 60 m above ground to as much as 200 m. FQM contracted Geophysical Surveys and Systems (GSS) to do extra micro-levelling to make useable channel images. This extra work demonstrated that the EM could pick out anomalies which coincided with domes.

9.3 Controlled-Source Audio Magnetotellurics (CSAMT) survey

FQM contracted GSS in September 2009 to conduct a Controlled Source Audio-frequency Magnetotellurics survey (CSAMT). Four east-west orientated CSAMT sections were surveyed to the south of Kansanshi Main Pit as shown in Figure 9-1. The main objective of the survey was to image the sulphide vein swarms which strike north-south through the Main Pit. A station spacing of 25 m was used along the lines. GSS undertook four types of inversion modelling on the data, mostly starting with a one dimensional Bostik inversion and then proceeding to various types of two dimensional inversions.

Two outcomes from this survey are that the CSAMT survey did not pick out the vein swarms but it did map the graphitic phyllites.

9.4 Airborne Magnetic and Radiometrics survey

In November 2010, a high-resolution aeromagnetic survey was undertaken over 118.5 km2 of the Kansanshi licence area (Figure 9-1) by New Resolution Geophysics (NRG) using their Xplorer horizontal gradient magnetometer system on a helicopter boom with 16L radiometric crystals. The flight height was 30 m on 100 m spaced N-S oriented lines. Magnetic levelling and Noise Adjusted Singular Value Decomposition (NASVD) radiometric processing were conducted by NRG.

The survey resulted in good quality, high resolution aeromagnetics and radiometrics images that have been used to refine the surface geology of the Kansanshi mining licence.

9.5 Induced Polarization survey (IP)

In August and September of 2010, GSS undertook a gradient array IP survey over the South East Dome, over an area of 1.2 km^2. The objective was to test the applicability of this method in mapping out the veins which host copper mineralisation.

The survey was carried out using a 6-channel Zonge GDP32II receiver and a Zonge GGT10 transmitter. The potential spacing was 10 m. A current flow of 2 amps or more was transmitted into the AB electrodes. All data was recorded at 0.125 Hz and a standard Newmont Window was used to calculate the chargeability. Positional data was recorded with a handheld GPS. All coordinates are in UTM 35S, Arc1950.

Once the gradient array survey was complete, two lines of pole-dipole IP were carried out. The data was quality-controlled by GSS using software supplied with Zonge hardware. The main outcome from this IP survey showed resistivity was mapping the shape of the South East Dome. One major drawback was the length of time taken to complete this type of survey. The method did not successfully map out the veins hosting the mineralisation.

9.6 Downhole geophysics surveying

Six holes were geophysically logged downhole by Gap Geophysics (QuikLog), based in Johannesburg, in 2009. The objectives of this work were two-fold: to measure and record physical properties for the various rock-types present in order to make informed decisions about proposed future work; and secondly to ascertain which surface geophysical techniques would be most suitable for direct detection of the mineralised veins. The following holes were logged:

- KXD55 – top 152 m
- KXD62 – top 148 m
- KXD68 – top 242 m
- KXD71 – top 116 m
- KXD72 – top 150 m
- KXD75 – top 406 m

It was observed that the carbonaceous phyllite responds strongly in terms of both IP and conductivity where disseminated sulphide volume is approximately above 2%. Large veins tended to show a broad EM response but did not show an IP response. It was concluded that EM was most suitable for detecting mineralisation.

9.7 Sub-Audio Magnetic (SAM) survey

In 2011, GAP Geophysics of Australia conducted a Sub Audio Magnetic survey (SAM) over 4.7 km^2 to the southeast and northwest of the Kansanshi mining licence as displayed in Figure 9-1. The objective was to compare a standard IP survey to an SAM survey to see which of these offered the most detail and geological information. SAM surveys yielded three outputs: magnetometric resistivity data (MMR), total field magnetics and EM data.

Results were variable. MMR results did not display the resolution of the IP survey from 2010 but picked out trends on the northwestern block which were similar to trends from the airborne magnetics. Magnetics data from the survey showed very high resolution results but was comparable to the airborne total field magnetic results from 2010. Results from the EM output of the survey did not show an improved resolution over the airborne EM survey conducted in 2007.

9.8 Audio Magnetotellurics (AMT) survey

A 1,250 m long trial Audio-frequency magnetotellurics (AMT) survey was conducted over the South East Dome in July 2011 by GSS. This survey used a Zonge GGT32 II receiver and AMT coil with E dipole length / spacing of 50 m and H spacing of 100 m. The line was oriented at 055^{o} and was designed to test the full extent of the South East Dome. The data was processed with Zonge MTEdit version 3.0 and inverted with GSS SC2D software version 3.2.

The AMT image showed excellent correlation with the geological interpretation which is based on drilling and more work is planned. Results were not significant for revising the geological interpretation, but were sufficiently encouraging for more detailed work to support an improved geology model of unmined areas and for future exploration.

ITEM 10 DRILLING

The following information is a summary of available information as used in this Mineral Resource estimate update and focusses on the new drilling since the previous 2012 Technical Report as collected by the issuer.

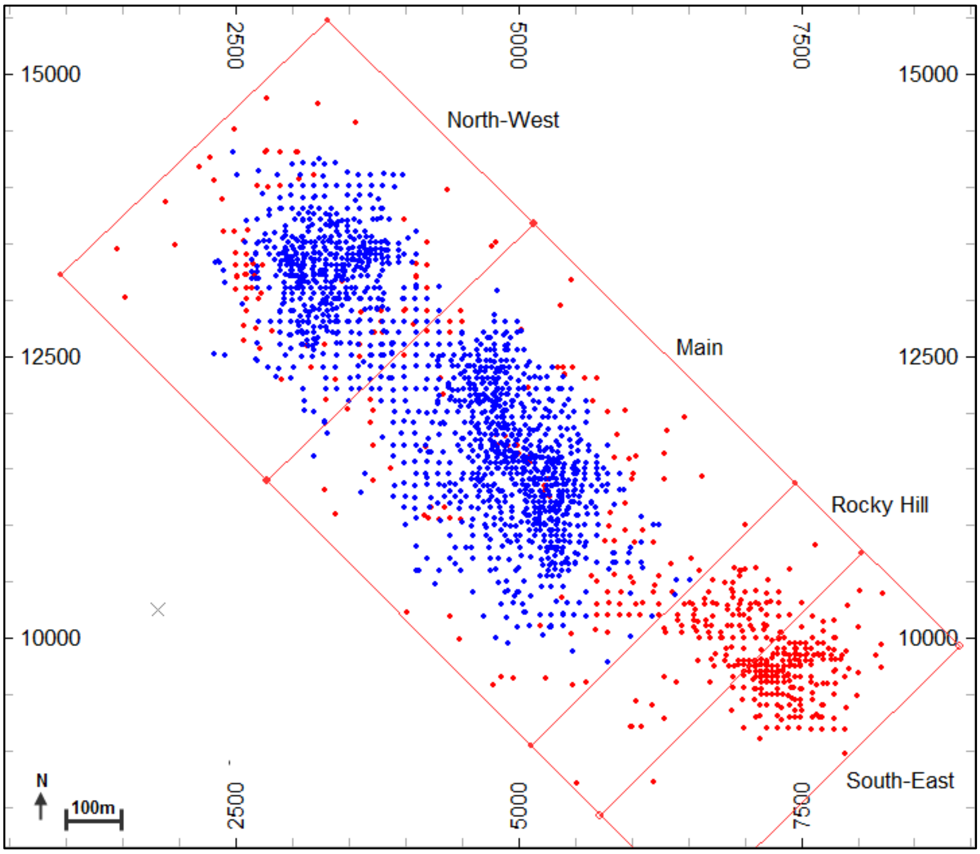
10.1 Diamond drilling

Since 2009, there has been ongoing Mineral Resource definition diamond drilling undertaken by the issuer. Resource definition drilling around Main, North West and South East deposits was run by the issuer's mine geology and exploration teams.

The majority of drilling started at a 100 m grid spacing, with infill drilling to a 50 m grid spacing. Key target areas since the 2012 Mineral Resource estimate include extensional and infill drilling of the Main and North West deposits as well as covering the South East Dome and Rocky Hill areas (Figure 10-1).

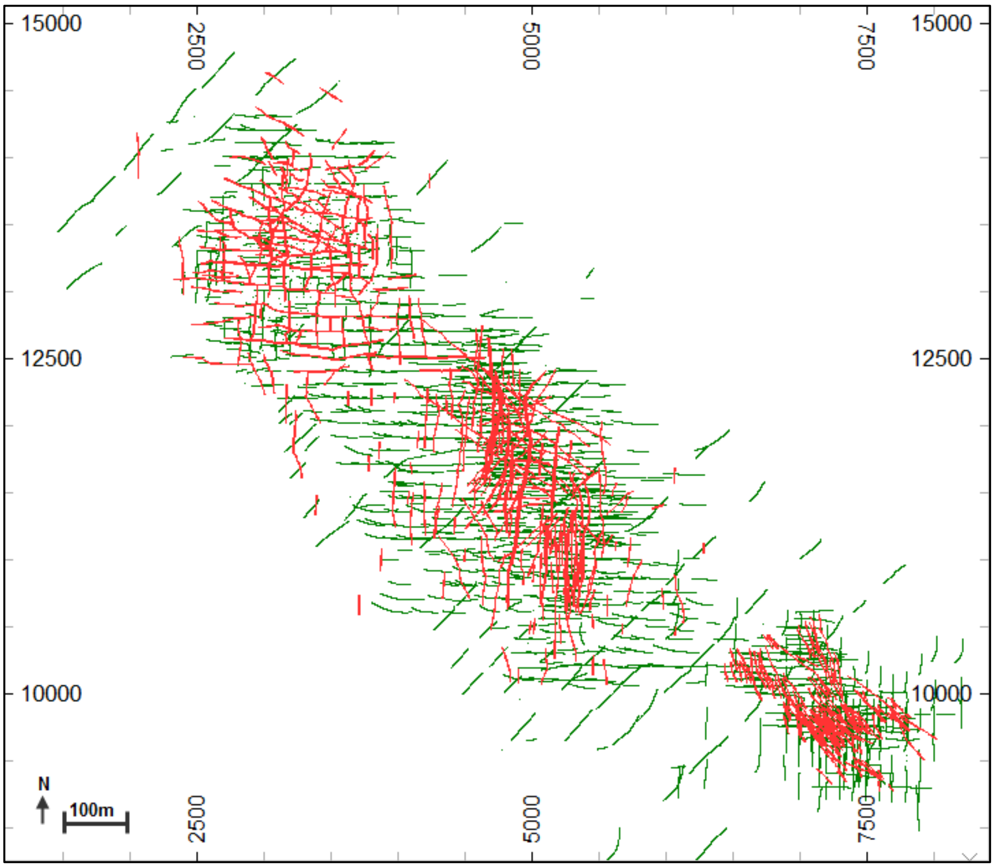
This Mineral Resource estimate update has benefited from an additional 332 new diamond drill holes logging and sampling data since the previous 2012 Technical Report. The additional drilling represents an extra 138,693 m of core and over 100,000 new samples and analytical results.

Figure 10-1 Kansanshi diamond drillhole coverage with new holes coloured red.



Diamond drilling has focussed on defining the extents and continuity of geology and mineralisation in sub horizontal strata and sub-vertical veins. As such multiple drilling directions dipping between 60 to 70 degrees were used to ensure coverage of relevant geology and sampling detail. Drilling directions and angles were guided by prevailing antiform strike and dip as well as dominant vein orientation as known from pit mapping. This approach has maximised the angle of intersection with vein and strata mineralisation as well as provide for infill and coverage required to define continuity and extents of mineralisation. In Main Pit, veins are generally trending N-S / NNE-SSW. Drilling in Main Pit is oriented E-W to intersect the N-S veins and structural domains. There is a high grade core to the centre of the Main Pit deposit where all stratigraphic units are mineralised including into the Lower Pebble Schist unit. North West Pit has three dominant vein orientations: N-S; E-W; and oblique. Drilling in North West Pit is in both N-S and E-W direction to intersect the multiple vein orientations. Regional drill holes are oriented in the N-E direction, perpendicular to the Kansanshi Antiform structure (Figure 10-2). At South East Dome, initial drilling was focused on defining the shape and extent of the dome. Holes were drilled primarily on a 100 m x 100 m grid, with holes oriented both N-S and E-W. Infill drilling to 50 m x 50 m was focused on the high grade zone at the centre of the dome. Sample lengths ranged from 0.5 to 1.5 m in length according to the geology and mineralisation boundaries. The significantly mineralised veins were generally in excess of one metre wide and strata mineralisation widths range from a few meters to 10's of metres.

Figure 10-2 Drill orientations at Kansanshi and South East Dome



10.1.1 **Core recovery analysis**

All diamond holes drilled for Mineral Resource purposes since January 2010 have core recovery and RQD data that were routinely collected during diamond drilling. At the request of KMP in 2012, CSA conducted an investigation of recovery data in order to asses any risk of recovery bias with regards to drill contractor, weathering profile and/or grade.

CSA identified a number of recoveries in excess of 110% with more than 60 having values over 200%. These were specifically from historic holes, for which the quality of original recovery data was not able to be verified.

Analysis of core recovery by drilling contractor highlighted no significant biases with global recovery averages close to 88%.

The recovery analysis by weathering profile similarly showed no significant issues. Poor recoveries in weathered material were to be expected with soils and laterites tending to lower values. At Kansanshi, soil and laterite material had recoveries of 82.3% and 77.7% respectively and at South East Dome they were 73.7% and 74.8% respectively. Low (43.7% average) recoveries were noted for material having no weathering code and were confirmed to be due to solution cavities in the shallower carbonate lithologies. CSA recommended that further work be completed to verify these intervals as they often coincide with high grade intercepts. The low recovery intervals represent 4 % of the data and were mostly located in mined out areas.

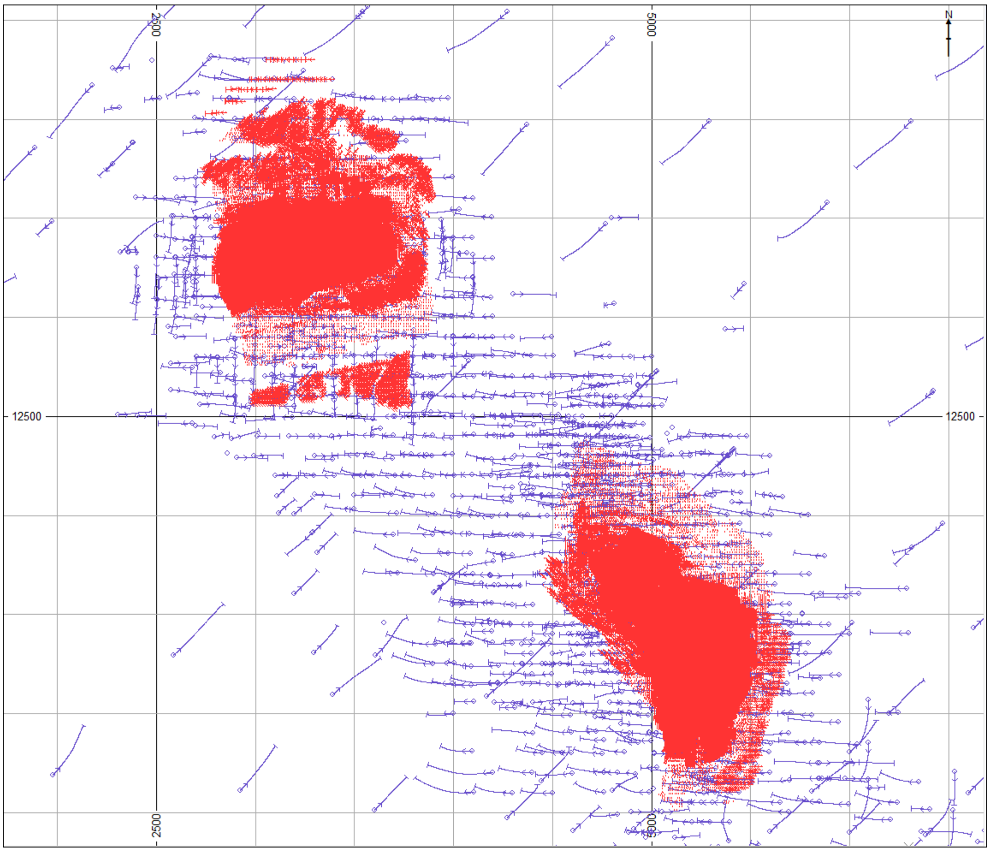
Investigation of the degree of correlation between recovery and TCu or AsCu highlighted no correlations.

10.2 **Grade control drilling**

Grade control reverse circulation (RC) drilling was completed by KMP on a 10 m (north) by 12.5 m (east) grid at a dip of 60^{o} and generally to a depth of 51 m. This corresponds to a vertical depth of 40 m, or the equivalent of four 10 m high mining benches. Hole depths vary from 33 m to 99 m and are a function of location in the pit and prevailing geology. Azimuths vary between 90^{o}, 220^{o} and 270^{o}, depending on the prevailing geology. RC drilling has been used for quality sampling and chip logging in order to provide mine planning with detailed geology data ahead of time. A comprehensive QAQC program was implemented early in 2013 providing for accurate and precise sample results with demonstrated control on contamination for the RC drilled samples. At the time of this Mineral Resource estimate update only half of the available RC drilled holes were logged and therefore used in this estimate. Figure 10-3 below highlights the relative position and distribution of grade control RC drilling.

Grade control RC drilled data for Main and North West deposits was previously not included in the estimation of Mineral Resources. In 2013, following a review of data by FQM and CSA, it was deemed relevant to include this significant dataset into the Mineral Resource estimate update reported herein. The data consists of inclined RC drilling with a total of 17,702 drilled RC holes representing 895,948 m and approximately 358,000 samples.

Figure 10-3 RC grade control drilling (red) across North West and Main pit areas relative to diamond drilling (blue).



10.3 **Collar surveys**

For collar survey data prior to 2010 refer to Hanssen *et al.* (2010). Since 2010, collars have been surveyed by the mine survey department using a Differential Global Position System (DGPS) and/or Total Stations. Conditions requiring a combination of the two include cloud cover or thick forest, where the DGPS will struggle to fix a position. The DGPS has an accuracy of 0.02 m in the x-y direction and 0.05 m in RL, while the Total Station accuracy exceeds this.

The UTM co-ordinate system is calibrated for the entire mine license, while the Local Grid calibration was designed to cover the two operating pits. As such for the Mineral Resource work, focused in and around the operating pits, both co-ordinate systems are accurate. The drilling on South East Dome and Rocky Hill was originally recorded in the UTM system due to the distance from the survey base station. This also includes holes drilled at more than 1 km from the pit area.

Table 10-1 lists the transformation parameters used for converting UTM Zambia grid coordinates to Kansanshi Mine grid coordinates. Both co-ordinate systems are stored in the database. From these points a Surpac macro calculates the translation and rotation prior to import into the drill database.

Table 10-1 Transformation parameters for Kansanshi Mine Grid and UTM Zambia

	MINE LOCAL GRID		MINE UTM GRID	
Point ID	y	x	y	x
A	14563.906	1269.213	8666711.922	434897.608
B	15472.174	7621.027	8666501.713	441305.479
C	11316.742	4878.666	8662889.489	437884.870
D	7152.654	7216.941	8658385.753	439461.968
E	6023.460	5068.826	8657647.990	437152.030

10.4 **Down hole surveys**

Only diamond drill holes are surveyed using the Reflex EZ-TRAC tool. RC grade control holes were not surveyed down the hole due to their limited drilled depth and very low risk of deviation. Surveys were taken every 50 m, beginning at 25 m, and at the end of the hole. The instrument measures magnetic azimuth which is converted to Mine Grid for use in the Mineral Resource estimation. The surveys were taken by the driller and recorded at the drill rig. The supervising geologist signs off on all surveys. Surveys were corrected for magnetic declination as per Table 10-2

Table 10-2 Magnetic declination correction

Magnetic North	National	Local
Recorded at rig	= Mag north - 4.5^{o}	= Mag north - 14.5^{o}

10.5 **Core orientation**

Orientated drill core has been used in the Kansanshi exploration drilling programme since the beginning of 2012 and orientation tools are currently in place on all drill rigs. Structural measurements are taken using a Goniometer, in alpha and beta angle format and converted to dip and dip direction using a Core Solutions (Scott *et al*; 2005) MSExcel based programme for determining the real-space orientations of planar and linear fabrics in axially-orientated core.

Orientation of core was in place throughout the project on the rigs drilling on South East Dome. Orientation of core was done using the Reflex ACT II RD tool. Geologists used a “rocket launcher” to orientate the core and a compass to record dip and direction of the structure.

ITEM 11 SAMPLE PREPARATION, ANALYSES AND SECURITY

The following information was summarised from the 2012 Technical Report (Titley, CSA, 2012) and updated according to current practice relevant to this report. Additions include RC grade control sampling analysis and security.

11.1 Core and chip logging

Core from Kansanshi and South-East Dome drilling was logged at the site core shed. Drill core was logged for lithology, mineralisation, alteration, weathering and structure by a team of qualified geologists. Logging was recorded directly into a MSExcel workbook template on notebook computers. The template has a set of pre-determined database library codes which can be selected through a series of drop down menus. Free cells have constraints attached to them to prevent, as far as possible, the recording of incorrect information. Any modifications to the standard codes were discussed within the geology team and actioned by the database administrator.

Logging intervals range from 0.5 m and were recorded to two decimal places. Changes in lithology, mineralisation, weathering and alteration were recorded as intervals. A logging manual outlines all relevant codes for lithology, alteration, weathering and mineralisation (Kansanshi plc, 2012). Wet uncut core was photographed with information boards recording the hole, tray number and core interval. Logging data sheets were loaded into the SQL database through a DataShed front end by the DBA. Electronic data loading eliminates manual data translation errors.

RC grade control chip logging of the washed chips (in trays) was completed at the in-pit drill site. Tray intervals are as per the 3 m sampling interval and logging data is restricted to key lithology types. The same logging templates as per diamond logging were utilised to record the lithology data.

11.2 Core and RC chip sampling

After logging, all diamond core was marked out for sampling and was routinely done from top to bottom of each hole. Sample lengths range between 0.5 m to 1.5 m according to mineralisation and geological boundaries. High grade copper intervals (greater than 3% Cu) were flagged, as well as the rare intervals with native copper. QC samples were inserted routinely throughout the sampling process. All Kansanshi drill core was sampled for analysis of total copper, acid soluble copper and gold. Marked drill core was cut with a diamond saw; one half was submitted to the laboratory for analysis with the remaining half retained in the labelled core tray and securely stored on site. Holes within the regional drilling lines were only assayed for total copper.

Field RC chip samples were taken via a levelled on-rig cone splitter which delivers a homogenised 12 kg sample mass representing each percussed 3 m. The RC field sample is split further with a riffle Jones splitter to a 3 kg mass which is then bagged in a numbered bag and delivered to the on-site laboratory sample preparation facility.

11.3 QAQC insertion

Certified Reference Material (CRM) standards were inserted into the sample numbering sequence at a frequency of one in 50 samples. Standards have pre-prepared sample numbers so the logging geologist needs only record the details in the sampling spreadsheet template. RC chip samples used a CRM insert rate of 5%.

Diamond drilled coarse crush duplicates were taken on average every 25 samples down the hole. They were inserted as part of the sample sequence with the duplicated sample number inserted into the ORIG_SAMP column of the sampling file. Duplicated samples were assigned by the logging geologist and where practicable were chosen from higher grade intervals. RC chip field duplicates were taken every 20th sample as a rigid routine. Pulp duplicates were inserted on average every 25th sample for both Diamond and RC chip sampling. The QP notes that RC QAQC was only implemented from February 2013 and that previous RC data quality cannot be assumed. The impact of the RC samples with no QAQC is however believed minimal due to them being located in mined out areas distal to the remaining declared Mineral Resources.

Blanks were inserted at the end of a holes sampling with additional inserts following high-grade intervals in order to monitor contamination.

CSA undertook a database review and an audit of the QAQC data and no fatal flaws were noted (FQM, 2012[2], FQM, 2012[3]). Results are summarised below:

- The QAQC programme implemented by FQM at Kansanshi for the diamond core samples was rigorous and comprehensive. Sufficient QAQC material has been included with the drill samples to give confidence that assay results obtained should accurately reflect the samples. Ongoing monitoring occurs with monthly QA/QC reports generated in QA/QCR; issues were investigated timeously and although some issues were ongoing, no fatal flaws were apparent.
- Some of the Exploration assay results prior to May 2013 might not be representative of the samples as failed CRMs were re-assayed, but not the associated samples (from the same digestion batch). This could potentially mask the fact that there were issues with the QAQC and samples as the corrected re-assayed values for the CRMs have been included in the database. QAQC procedures were updated in May 2013 which appears to have addressed this issue. However, enough checks have been undertaken to ensure that the assay results achieved should be representative of the samples used.
- The grade control samples were analysed at the Kansanshi mine laboratory which is not accredited. QAQC checks are a relatively recent addition in the grade control drilling with no effective QAQC undertaken prior to February 2013. Thus any grade control results prior to this date would have to be viewed with caution.

11.4 **Laboratory sample preparation**

The cut, bagged and numbered core were delivered to the onsite ALS Chemex (ALS) managed sample preparation laboratory. RC bagged and numbered chip samples were delivered to the on-mine core shed and were further prepared by the on-mine RC technical staff. Sample preparation starts with drying core samples at 120°C +/- 20°C, and then jaw crushing the core to a minimum 2 mm crush size. RC chip samples were only dried if wet and then crushed. Crushed samples were then repeatedly split to the required 1 to 1.5 kg sample mass which was then pulverized to at least 85% passing 75 µm. The 1 kg crushed sample was pulverised using a ring mill pulveriser with a chrome steel ring set. The pulp sample was transferred onto a rubber rolling mat and then homogenised by repeatedly rolling the pulp back upon itself. An approximately 150 - 250 g pulp sample was scooped from the 1 to 1.5 kg pulp and packed into a numbered envelope ready for analysis.

11.5 Laboratory analysis

Laboratory analysis for diamond core samples were analysed for total copper (TCu), acid soluble copper (AsCu), cyanide leach copper (CnCu) and gold (Au). In addition, approximately 10% of diamond core samples were analysed for multi-element data. Diamond core samples were analysed by the on-site ALS managed Chemex laboratory (non-accredited) and the ALS Chemex Johannesburg laboratory (accredited). While the on mine ALS Chemex laboratory is not accredited, KMP exploration and mine geologists have assured quality analyses with comprehensive QAQC, which includes umpire check analysis at the accredited Genalysis laboratory in Johannesburg. RC grade control chip samples were analysed for TCu, AsCu and gangue acid consumption (GAC) at the on-mine managed KMP laboratory. Every fourth RC holes' samples were sent to the ALS Chemex laboratory for TCu, AsCu, CnCu and gold analysis. The pulp reject samples were stored at the ALS Chemex laboratory according to KMP instructions or in the case of the RC pulps, stored at the mine core shed facility. Laboratory visits were regularly completed by the KMP mine and exploration geologists as well as ad-hoc visits by external consultants. The intention of these visits is to ensure optimal sample preparation and analysis through pro-active observation and communication.

Diamond core samples were analysed at the on-site ALS managed Chemex laboratory (independent of the issuer) where:

- Total copper was analysed using a four acid digest and an Inductively Coupled Plasma with Atomic Emmission Spectrometry (ICP-AES) finish
- Acid soluble copper was analysed using a single sulphuric acid leach and an Atomic Absorption spectrometry (AAS) finish
- Cyanide soluble copper was analysed using a cyanide leach and an AAS finish
- Gold was analysed using a fused precious metal bead which is then digested in acid and analysed with AAS against a matrix-matched standard.

RC chip samples were analysed at the on-mine managed KMP laboratory where:

- Total copper was digested with a four acid solution and analysed with AAS finish
- Acid soluble copper was digested with a single acid leach and an AAS finish
- Gangue acid consumption (GAC) was measured using standard volume titration techniques.

11.6 Comments on sample preparation, security and analytical procedures

It is the opinion of the Qualified Person, Mr David Gray that the sample preparation, security and analytical procedures of the Kansanshi diamond and RC chip samples are of good industry standard. Sample collection and preparation have been practiced with standard equipment suitable for ensuring that samples are representative of the in-situ geology. Field core and RC chip samples are stored on-mine in secure, locked facilities. Appropriate delivery and sign-off tracking documents were used for delivery of samples to the respective laboratories. Laboratory analytical methods and techniques are standard to the industry. Combined with QAQC, these have assured that sample analytical results were representative by ensuring sample precision, analytical accuracy and by containing contamination. Results were believed to be adequate for supporting this Mineral Resource estimate update of the Kansanshi deposits.

ITEM 12 DATA VERIFICATION

The Qualified Person, Mr David Gray, has visited the Kansanshi mine at least twice a year over the last two years. During these visits Mr Gray has verified the respective aspects of data collection associated with the diamond drill core and RC sampling as well as the respective QAQC. In addition, Mr Gray has verified the in-pit mapping and use of sufficient quality data together with verifying use of robust methods for geology modelling and resource estimation.

In addition, the contributing author, Mr Malcom Titley has supported verification of the integrity of diamond core sampling, bulk density estimation, sample preparation and despatch procedures and in-pit mapping. Verification procedures applied by Mr Gray together with comments on limitations include:

- Site visits were used to verify drilling, sampling, preparation, analysis and QAQC practices.
- RC grade control QAQC practices were implemented from February 2013 and are currently ongoing with recommended improvements. Accordingly, RC data pre-2013 were noted to be at risk from having no supporting QAQC. However, the QP notes that these samples were located in mined out areas that were distal to the current resource estimates. As such, these samples would have limited to no impact on current resource estimates.
- Diamond core sampling QAQC was verified as comprehensive and sufficient to support precise and accurate results with good evidence to demonstrate that contamination was contained. The QP notes that prior to May 2013 that batches from failed CRM's were not re-assayed, however due to other CRM's passing, these batches were accepted.
- Bias risks between samples using different drilling methods and field sample masses were investigated by the QP with some risk of bias identified between the RC and diamond drilled samples. Samples used in these comparisons were however not spatially twinned. As such further studies need to be completed in order to quantify the degree of this bias.
- In-pit observation of prevailing geology was confirmed as been adequately reflected in the current geology model. Current detailed structural, lithological and vein mapping detail was not incorporated into this updated estimate due to its incomplete and complex nature. Future estimates may benefit from this improved geological detail.
- Diamond and RC drill collar coordinate measurements and visual checks against digital data and database data were confirmed.
- The documented sampling and sampling preparation methods were investigated and verified against data stored in the database.
- Verification of a small percentage of actual laboratory assay records against database data.
- Verfication of improved data validation procedures as utilised within the SQL Datashed database. The grade control Datashed database was noted to have several duplicate coordinates. Duplicate holes were identified and removed prior to Mineral Resource estimation.
- The majority of diamond core analysis was completed at ALS Kansanshi and ALS Johannesburg with umpire checks done at Genalysis Johannesburg. The Johannesburg laboratories apply ISO approved procedures and management certificates. Certification is required by the ALS Kansanshi laboratory and KMP on-mine laboratory. Quality of sample results needs to be verified through further comparative studies.

- The Kansanshi Exploration data and the grade control data are hosted in SQL databases, ensuring that data is validated as it is captured and exports are produced regularly. Assay results are merged into the database from the laboratory certificates preventing transcription or mapping errors from occurring. The SQL database was provided by FQM and reviewed by the CSA data manager and no significant issues were observed.

It is the opinion of the QP, David Gray, that the available data is adequate for use during this Mineral Resource estimate. The data has sufficient coverage of the respective domains of mineralisation and has been collected, sampled and analysed for storage in a safe a secure manner. Sample assay data is believed of adequate quality from use of relevant sampling methods and good QAQC practices. Continued RC grade control QAQC and control of any bias risks, together with enhanced data validation will serve to increase confidences in future data and the resulting estimates.

ITEM 13 MINERAL PROCESSING AND METALLURGICAL TESTING

The following information is reproduced in part from the December 2012 Technical Report (Journet and Cameron, DumpSolver, 2012), with an update provided by Robert Stone (QP).

13.1 Overview

The mineral processing facilities at Kansanshi have undergone several periods of expansion since commissioning in 2005. The details of expansions pre-2013 are documented in the December 2012 Technical Report (Journet and Cameron, DumpSolver, 2012) and are summarised in Item 17.1.

The current capacity of the processing facilities is 15.6 Mtpa of float/leach ore and 13.2 Mtpa of sulphide ore, arising from more recent expansions as described below.

13.1.1 Leach expansion and smelter background

The 2013 leach expansion included the construction of additional flotation, leaching, CCD thickeners, SX and EW facilities to increase the cathode production by boosting the treatment rate to 12 Mtpa of leachable ore.

Milling of the additional ore tonnage has been achieved by converting an existing milling circuit to handle oxide ore in conjunction with the installation of a new (larger) oxide crushing plant.

The leach expansion coincided with the construction of a smelter on-site at Kansanshi. The new smelter will produce large volumes of sulphuric acid as a by-product, which will be consumed by leaching of the oxidised ore in the expanded plant. The reduced cost of the smelter acid allows a significant proportion of ore previously classified as mixed to be economically treatable by leaching.

The rationale behind the construction of a new smelter at Kansanshi arose from the current shortage of smelting capacity in Zambia (exacerbated as the Company's Sentinel Project comes on line) which could constrain the sale of concentrate from the new S3 plant (Item 13.1.3), once completed.

13.1.2 S3 sulphide expansion project

The S3 sulphide expansion project is an expansion, in two phases, to the existing Kansanshi processing facilities. The expansion includes the construction of a new copper concentrator capable of treating an additional 25 Mtpa of sulphide ore.

The design for the S3 expansion has been based on that previously designed for FQM's new Trident Project to treat ore from the Sentinel Mine, although taking into account metallurgical differences and site specific conditions.

A design throughput of 25 Mtpa (the capacity of one of the two processing trains at the Company's Trident Project) has been adopted for the S3 expansion and the following fundamental differences from the Trident flowsheet are noted:

- Kansanshi sulphide ore has a free gold content requiring gravity recovery.

- Kansanshi sulphide ore does not require flash flotation.
- There are no gangue minerals requiring depression.
- A shorter flotation residence time and two-stage cleaning will suffice. The only reagents required will be collector and frother.
- Concentrate will be pumped approximately 4 km to the filtration plant at the new Kansanshi smelter in place of trucking.

13.2 **Test work results and process recoveries**

The ore selection at Kansanshi is governed by the relative proportions of acid soluble copper (ASCu) and acid insoluble copper (AICu) in the ores, where total copper (TCu) equals ASCu + AICu. The ore is classified into three ore types based upon the AsCu/TCu ratio.

1. Sulphide ore – defined as ore dominated by primary sulphide minerals. The ASCu/TCu ratio range is less than 0.1, as oxidation has a detrimental impact on recovery.
2. Mixed float ore – defined as ore with an ASCu/TCu ratio ranging between 0.1 and 0.5, which is dominated by primary and secondary sulphide minerals with minor acid leachable minerals. In the Mineral Resource model this material is referred to as Mixed.
3. Mixed leach ore – defined as ore with ASCu/TCu ratio greater than 0.5, which is dominated by primary oxide minerals with minor secondary minerals. In the Mineral Resource model this material is referred to as Oxide.

The three ore types described above broadly correlate with materials in the three geological weathering states (completely weathered, partially weathered, and fresh). A fourth material category referred to as “refractory” in the Mineral Resource model cannot yet be treated economically and is stockpiled.

The sulphide ore is treated by conventional flotation. The mixed float and the mixed leach ore types are both treated by flotation to recover a proportion of acid insoluble copper minerals, with the mixed float flotation step including CPS (ie, conditioning with NaHS) to further enhance the mineral recovery. Tails from the mixed float are directed to final tails whilst tails from the mixed leach are directed to leaching, followed by SX and EW to produce copper cathode. The reduced cost of acid from the smelter allows a significant proportion of ore previously classified as mixed float to be reclassified as mixed leach and be economically treatable by leaching. As such, the transition between mixed float and mixed leach ore remains flexible, based upon economics and ore availability.

Recovery parameters are based on KMP’s analysis of actual production. The detailed analysis shows that the recoveries for each of the process routes are non-linear. These recovery equations have been reviewed and the mixed leach copper circuit recovery formula updated from that presented in the 2012 Technical Report (Journet and Cameron, DumpSolver, 2012). The new and old equations are set out below.

13.2.1 **Sulphide circuit copper recovery**

Sulphide copper circuit recovery will vary with the total copper (TCu) head grade and the proportion of acid soluble copper (ASCu). Test work concludes that if acid soluble copper is less than 0.05% ASCu and total copper is greater than 0.35% TCu, then copper recovery varies from around 84% up to 93% if total copper exceeds 0.75% TCu. The equations to model this are complex and are based on the two-product formula. Essentially, recovery is shown to be a function of the acid soluble to total copper ratio and the copper grade.

$$Sulphide\ Recovery = \left(\frac{(TCu_{feed} - TCu_{tails})}{(26.5 - TCu_{feed})} * \frac{26.5}{TCu_{feed}}\right)$$

Where:

$$TCu_{tails} = (0.0329 * TCu_{feed} + 1.5132 * ASCu)$$

26.5 (%) = the concentrate grade

From the equation referred to in Journet and Cameron (DumpSolver, 2012), the only change has been to the concentrate grade, which was previously 27%.

13.2.2 **Sulphide circuit gold recovery**

With the additional Falcon concentrators, overall gold recovery averages 63.4% with 21% of this reporting as gold ore via the gravity circuits and the remaining 79% as gold in concentrate. These recovery factors are unchanged since 2012.

13.2.3 **Mixed float circuit copper recovery**

Predicted mixed circuit recoveries use a similar two-product formula approach as that used for the sulphide circuit. Recoveries can be less than 45% but may be as high as 80% depending on the mineralogy. Chrysocolla is not recovered by controlled potential sulphidisation (CPS) flotation.

Recovery is a function of the ratio of ASCu to TCu and grade. With higher proportions of ASCu to TCu, the recovery decreases. As the total grade increases, expected overall recovery also increases.

$$Mixed\ Recovery = \left(\frac{(TCu_{feed} - TCu_{tails})}{(26.5 - TCu_{feed})} * \frac{26.5}{TCu_{feed}}\right)$$

Where:

$$TCu_{tails} = (0.0688 * TCu_{feed} + 0.8 * ASCu)$$

26.5 (%) = the concentrate grade

From the equation referred to in Journet and Cameron (DumpSolver, 2012), the only change has been to the concentrate grade, which was previously 27%.

13.2.4 **Mixed float circuit gold recovery**

Overall gold recovery from the mixed circuit averages 67.7% for an average head grade of 0.2 ppm Au. The proportion recovered by gravity techniques is 43.4% with the balance of 56.6% reporting as gold in the mixed copper concentrate. These recovery factors are unchanged since 2012.

13.2.5 **Mixed leach circuit copper recovery**

Typically for mill feed having a 70% of total copper as acid soluble copper, the overall recovery is around 82% to 85%. Recovery of copper to concentrate (via pre-leach float) is 15% to 25%. Commensurate recovery to cathode is 68% to 58%.

Oxide recovery to concentrate is now modelled using the following equation:

$$Oxide\ Recovery\ to\ Concentrate = \frac{(0.035 * ASCu_{feed} + 0.40\ AICu_{feed})}{TCu_{feed}} * 100$$

Previously, the equation used in 2012 (Journet and Cameron, DumpSolver, 2012) was:

$$Oxide\ Recovery\ to\ Concentrate = \frac{(0.04 * ASCufeed + 0.45\ AICufeed)}{TCufeed} * 100$$

Oxide recovery to cathode via the leach/SXEW process is now as follows:

$$Oxide\ Recovery\ to\ Cathode = \frac{((0.92 * ASCu_{feed} * (1 - 0.035)) + ((0.45 * AICu_{feed}) * (1 - 0.4)))}{(ASCu_{feed} * (1 - 0.035)) * (AICu_{feed} * (1 - 0.4))} * 0.98$$

Previously, the equation used in 2012 (Journet and Cameron, DumpSolver, 2012) was:

$$Oxide\ Recovery\ to\ Cathode = \left(MIN\left(\frac{(0.6091 * 0.4066 + 120.992 * ASCufeed^{0.4043})}{(0.4066 + ASCufeed^{0.4043})}, 95\right) + AICufeed * 0.4\right) * 0.97$$

13.2.6 **Mixed leach circuit gold recovery**

Based on reported average head grades of 0.31ppm Au, the historical average overall gold recovery from the oxide circuit is reported as 64.5% with 75.3% of this reporting as dore from the gravity circuit and the balance of 24.7% in the oxide copper concentrate. These recovery factors are unchanged since 2012.

13.2.7 **Total acid consumption**

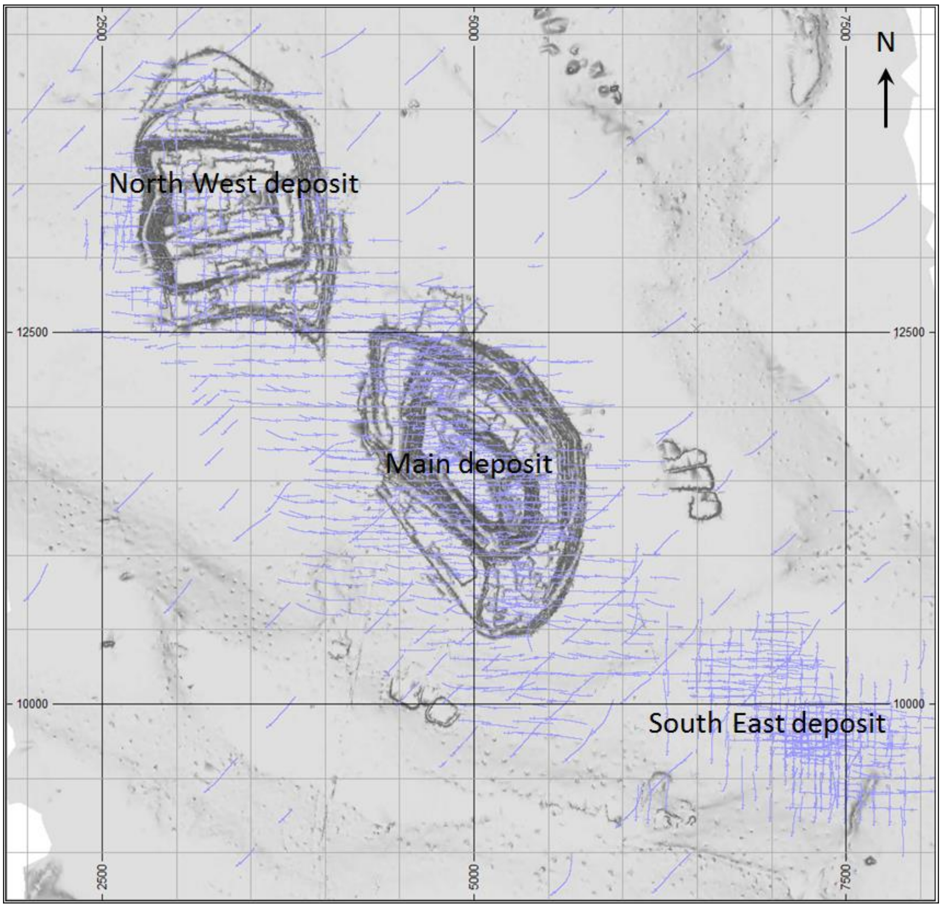
Total acid consumption is a function of the gangue acid consumption and acid soluble copper grade. The subsequent economic evaluation for the Mineral Reserve estimate assumes that a plentiful supply of sulphuric acid is available from the Kansanshi smelter and consequently acid consumption is not included in the cost of processing leachable ore.

ITEM 14 MINERAL RESOURCE ESTIMATES

Mineral Resource estimates have been generated for three vein hosted and strataform copper deposits within the Kansanshi mining area: these are the North West, Main and South East deposits. The Mineral Resource estimates were prepared under the supervision of QP, David Gray with the assistance of CSA consultants (led by contributing author, Malcolm Titley), and the KMP mine geologists.

The locations of these deposits are shown in Figure 14-1. The estimates were developed from a 3D block model containing the key geology domains and were based upon geostatistical applications using commercial mining software, Datamine Studio 3, as well as Surpac and Isatis.

Figure 14-1 Location of North West, Main and South East deposits at Kansanshi with pit surveys as at May 2015



14.1 Data

Since the 2012 Mineral Resource estimate (Titley, CSA, 2012), a total of 208 exploration diamond holes for a total of 82,498 m in the Main and North West, and 124 exploration holes for a total of 56,195 m in South East Dome (including Rocky Hill) have been included. This brings the total number of exploration diamond holes used in the Mineral Resource estimate update for this Technical Report to 1,982 holes for 537,204 m.

In addition to the exploration drilling, RC grade control drilling data was included in the estimation of the vein material where available (Main and North West). This dataset comprised 17,702 holes for 895,948 m and approximately 358,000 3 m length samples.

Data files were exported from Datashed as comma delimited text files. These text files were loaded into Datamine as Collar, Survey, Assay and Geology files. Validation comprised of checking for duplicate data, overlapping data, data with no valid collar, or collars with no valid data. Any errors encountered at this stage were discussed and addressed between CSA Data Management and FQM. Visual inspection of the magnitude of downhole deviation was completed in order to identify any improperly recorded downhole survey values. Assay data was also interrogated for any values outside of expected values.

Data was de-surveyed to create 3D assay drill hole files. All missing data was set to absent as FQM advised this data was either a cavity or a lost sample, and not waste. All negative data, reflected detection limit values and were set to half the detection limit per metal analysed and as per the analytical technique used.

14.2 **Geology model**

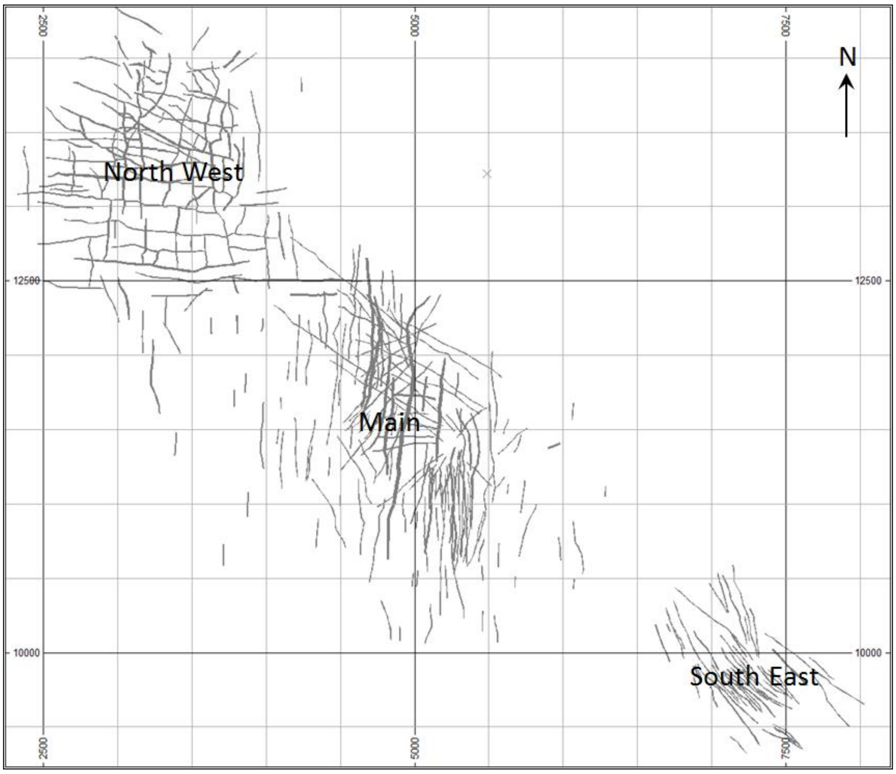
FQM provided CSA with geological wireframes used for domain boundaries within the estimation. These included stratigraphic boundaries, geological units, veins, weathering and the mineralogical oxidation profiles.

3D modelling was done in Surpac and Datamine. Depending on what was being modelled, either a digital terrain model (DTM) surface or 3D solid was created from joining strings and/or segments across drill line sections. Vein solids, residual and phyllite were extended by half drill fence spacing from the last drill hole intersection. The wireframe detail per geological feature was:

- Stratigraphy units modelled as surfaces.
- Intrusive gabbro unit on the eastern side of the Main Pit, generally barren with some mineralisation in the weathered zone. The gabbro was modelled as a 3D solid.
- Residual unit where weathering of the upper marble or top most marble by dissolution has resulted in a biotite-rich residual unit and which has seen supergene enrichment in some areas. Due to its high grade nature, and close proximity to fresh marble, which has a high Gangue Acid Consumption (GAC), it requires isolation for processing, and therefore detailed during modelling.
- Weathering profiles are based primarily on logged weathering and inform the specific gravity assigned to all stratigraphy units. They were not used as estimation boundaries during the estimation.
- Oxidation surfaces: The distribution of Cu, AsCu and Au grades are influenced by weathering, oxidation and lithology and are grouped according to an oxidation classification, namely: refractory (not processed), oxide/transitional and sulphide. The refractory domain is a leached zone which sits above the oxide domain. The sulphide domain is characterised by very low AsCu/Cu ratios, and displays a sharp boundary with the overlying oxide/transitional material. More detailed separation of oxide, transitional, and sulphide was completed post-estimation through the use of AsCu/TCu ratio ranges. The criteria used to wireframe the oxidation surfaces are tabulated in Table 14-1.

- Vein model wireframes were modelled as 3D solids for North West, Main and South East Dome (Figure 14-2).
- Topography wireframe which was provided by FQM, and which covers the full extent of the subject area (Figure 14-1).
- Pit wireframes which outlined the base of current mining operations at the end of May 2015 (Figure 14-1).

Figure 14-2 Plan view of the three Kansanshi deposits modelled 3D vein wireframe solids

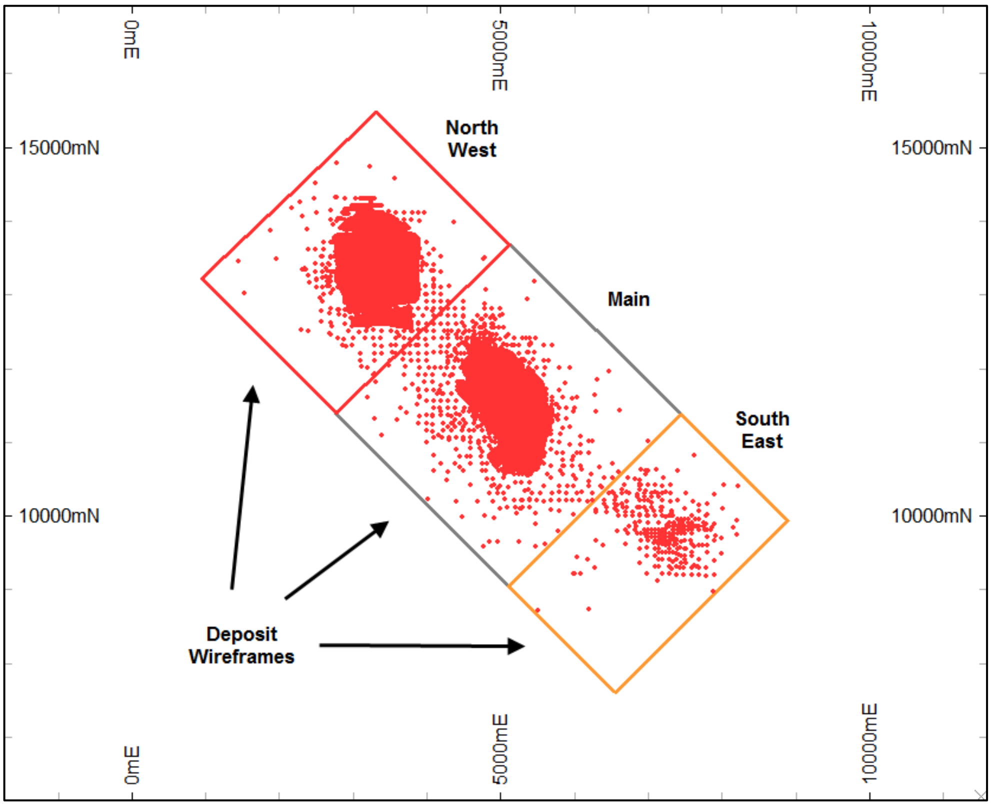


Deposit area wireframes which divided the Mineral Deposit estimation model into three deposits were provided to CSA by FQM and are shown in Figure 14-3.

Table 14-1 Revised Oxidation domain Parameters (FQM 20131)

Oxidation	Weathering Code	Common Mineralisation	TCu	AsCu	AsCu/TCu
Refractory	RES,LAT,SAP,SPR	Goethite, native Cu	Low (<0.5)	Trace	<0.15
Oxide (incl transition)	RES,SAP,SPR,FR	Carb (mal, az), Oxides (cup), Sulph (cc)	Mod - high	Mod – high	>0.15
Sulphide	SPR,FR	Sulph (cpy)	Mod - high	Low – trace	<0.15

Figure 14-3 Kansanshi deposit, showing data and deposit areas



14.3 Vein model

During 2013, FQM undertook a remodelling programme for the 3D vein wireframe solids per deposit. RC grade control data was included in order to increase the data support for the interpretation of vein wireframes. However, half of the RC grade control data lacked geological logging and as such vein volumes were restricted to Cu and AsCu grades. Wireframing methodology employed by FQM employed the use of close spaced data and a probability model to guide delineation of veins. This methodology and approach used was:

- veins were restricted to the MMC and UMC stratigraphic horizons, where vein development is most likely
- preferred vein orientations were known from pit mapping, dome or anticline location, with N-S and E-W being the most prevalent
- vein spacing increases and widths decrease with distance away from the dome or anticline axis and vein widths reduce vertically from top to bottom
- veins are sub vertical (or perpendicular to sub horizontal bedding)
- vein logged sample data from diamond drilling identifies a copper grade inflection of 1% above which sample grades are most likely to be associated with vein mineralisation
- only samples with assay values were considered and were composited to a 3 m length
- a categorical indicator model was kriged from sample data into an empty block model to highlight blocks most likely to be vein material
- the combination of these characteristics, sample and logged data and indicator models were used to define the centre line of each vein

- logged vein widths from diamond drilling were then applied to the centre strings and vein volumes were defined for the width of the MMC and UMC horizons (Figure 14-2)

It has been necessary to use the grade control data in order to interpret continuity in the veins. However, incomplete geological logging, and the use of grade data, may give the perception of vein thickening in higher grade stratigraphic units and may in fact not be vein mineralisation. Accordingly, only veins having widths greater than 3 m were modelled and widths were only guided by the true width information from diamond logged data.

Four vein orientation groupings were produced and a priority was applied according in order to eliminate the use of the same samples during estimation of the different vein groupings. The priority was as follows:

- North-South (N-S)
- East-West (E-W)
- North-West (N-W)
- North-East (N-E)

14.4 Data flagging

Raw assay data was flagged using the wireframes provided by FQM for geology, oxidation, weathering, pit, topography and deposit.

Vein wireframes were unsuitable for flagging of data, due to the uncertainty of their spatial location. Vein wireframes were deemed representative of vein volumes but do not accurately delineate vein position and therefore do not enclose all logged and/or assumed vein sample data. With this in mind, samples used for estimating grade into vein volumes were as follows:

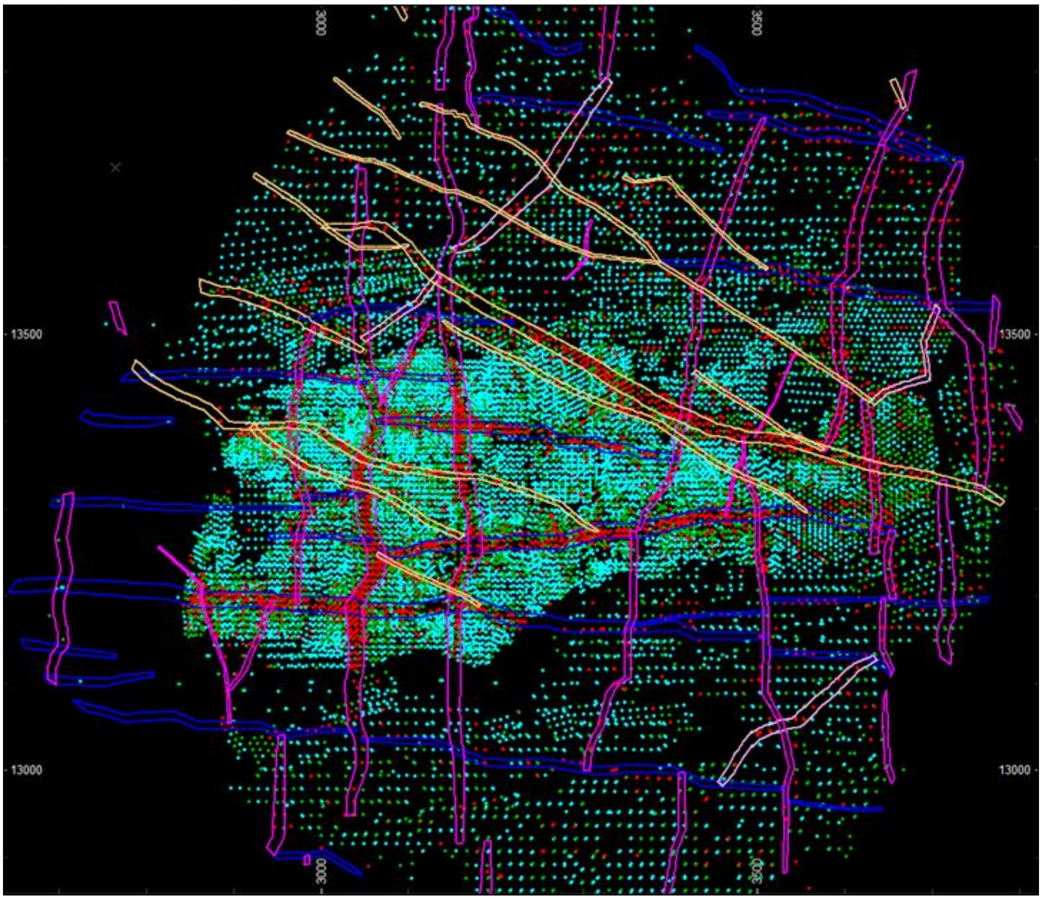
- Diamond vein data was identified by lithology logging (where LITH1=VN) and/or where logged Vein percentage was ≥60%.
- RC grade control vein data was identified using population statistics whereby Cu cut-offs of 0.9% and 1.1% were chosen for North West and Main Pit respectively. These cut-offs were chosen based on visual validation with vein continuity and the overall Cu grade distribution and statistics. There was a strong inflection in sample grade distribution at ~ 1% copper grade which visually coincides with vein volumes (Figure 14-4).

These vein data were then flagged using a modified nearest neighbour approach whereby:

- Vein samples (VEIN>0) were converted to 1 m x 1 m x 1 m block model cells in Datamine.
- A flagged, subcelled 20 m x 20 m x 10 m vein block model (coded by each individual Vein Number) was converted to a points file (using block centroids) and the Vein Number estimated (using nearest neighbour) into the Vein Sample 'blocks'.
- A Nearest Neighbour interpolation was completed whereby exploration data logged as vein within 20 m x 20 m x 5 m and grade control data within 5 m x 5 m x 5 m was flagged as VNGROUP1.
- Intercepts that met criteria 'a' and 'b' above but were not flagged by the above nearest neighbour approach (ie, were far away from the vein wireframes) were classified as VNGROUP2.

- Diamond data that was not logged as vein was flagged as VNGROUP0 and was therefore used in the stratigraphy grade estimate.

Figure 14-4 A sub horizontal section/plan view of North West veins compared to grade control samples. Samples below 0.8% copper (turquoise), 0.8-1.0% copper (green) and above 1% copper (red).



14.5 **Compositing**

Two compositing strategies were employed for the vein data (VNGROUP1) and stratigraphy data not proximal to interpreted vein wireframes (VNGROUP0 and VNGROUP2).

Due to the grade and thickness variability of the veins, plus the uncertainty around their spatial location on a local scale, it was deemed that average grades were a more reliable estimation of vein grade compared to localised estimates. Therefore, vein data was composited to single intercepts per vein, per drill hole, and statistics were produced on length weighted variables.

The remaining data (VNGROUP0 and VNGROUP2) was composited to 1 m. Though the dominant sample interval for drilling in stratigraphy was 1.5 m, 1 m was chosen as a compromise. 1 m composites provided better resolution across sub-horizontal geology boundaries, while it maximised availability of samples by reducing residual/rejection of sub-composite length samples.

14.6 Domaining and statistics

Data used in the vein estimate was domained by vein orientation and oxidation material. Data used in the stratigraphy estimate was domained by geology and deposit (i.e. stratigraphy / lithology unit). Vein numbers (501 to 504) were combined with oxidation numbers (termed in the model as REDOXN 1000 and 2000) to create 8 ESTZON units which were estimated separately (Table 14-2).

Table 14-2 ESTZON for vein model based on vein orientation and oxidation numbers

Geology	Vein	Sulphide	Oxide/Transitional
Veins N-S	501	1501	2501
Veins E-W	502	1502	2502
Veins NW-SE	503	1503	2503
Veins NE-SW	504	1504	2504

There were 32 estimation domains for the stratigraphy estimate (Figure 14-4). Some units do not extend into each deposit area, and were therefore un-estimated, though some small amounts of data may have existed in these units. These were: Main (TM, MPS), North West (UPS, TM, LD, PHY, DIO) and South East (PHY, LPS).

Statistics were produced to review the grade distributions of veins (Table 14 4) and stratigraphic units (Table 14 5). Histograms showed the distributions as positively skewed normal distributions. Swath plots show grade trends peaking in the domes of each of the deposits, and dissipating towards the edges. This information was used to inform the size of the search ellipse.

14.7 Top cuts

In order to restrict the influence of high grade samples, that may be real, but are not representative of the local grade distribution, top cuts were used to control the influence of outliers. Top cuts were performed separately on the datasets for the wireframed veins, un-wireframed veins, and the background strataform mineralisation.

Top cuts for the vein hosted mineralisation were in the ranges of 20% to 27% Cu, 10% to 14% AsCu, and 6 g/t to 12 g/t Au.

For the lower grade strataform mineralisation, top cuts were in the ranges of 0.2% to 29% Cu, 0.003% to 21% ASCu and 0.1g/t to 40 g/t Au.

14.8 Variography

Variography was completed on stratigraphic domains for Cu, AsCu and Au. There were moderate to strong correlations for Cu and AsCu, which varied depending on stratigraphic/geology unit. Correlations between Cu/AsCu were moderate to high, while correlations with Au ranged from low to moderate, for example a correlation coefficient of 0.55 in MMC was evident. To honour the correlations between the variables (where present), cross variograms were produced for use in the co-kriging algorithm.

Nuggets for Cu ranged from 19% in MPS to moderate (40%) in MMC and UMC to high (>60%) in LCS and LM. Ranges of first structures varied from domain to domain, and deposit to deposit, but averaged 30 m, with the vast majority of domains showing >80% variability within that short range structure. Longer range structures were in the order of 200 m to 300 m. Directional variography was modelled throughout, with first and second directions being omni-directional. Main rotation was horizontal towards 80^{o}.

Table 14-3 Estimation domains for the stratigraphic mineralisation

Geology Number GEOLN	Definition
199	Upper Pebble Schist (UPS)
200	Top Marble (TM)
201	Upper Mixed Clastics (UMC)
202	Upper Marble (UM)
203	Middle Mixed Clastics (MMC)
204	Lower Calcareous Schist (LCS)
205	Lower Marble (LM)
206	Lower Pebble Schist (LPS)
207	Lower Dolerite (LD)
208	Knotted Schist (KS)
210	Phyllite (PHY)
211	Residual, NW and Main (RES)
212	Residual, SE (RES)
213	Middle Pebble Schist (MPS)
300	Diorite (DIO)
900	Laterite (LAT)

Table 14-4 Sample statistics for the vein estimation

DEP	GEOLN	CU				ASCU				AU			
		Number	Minimum	Maximum	Mean	Number	Minimum	Maximum	Mean	Number	Minimum	Maximum	Mean
1 = Main	501	3 084	0.001	51.360	3.466	3 048	0.001	37.694	1.430	654	0.005	12.982	0.787
	502	161	0.003	20.521	3.294	157	0.001	11.718	1.381	36	0.005	2.790	0.435
	503	829	0.002	27.910	3.465	826	0.001	19.560	1.719	113	0.005	5.190	0.343
	504	623	0.004	33.450	3.100	618	0.001	15.000	1.402	64	0.005	2.419	0.540
2 = NW	501	1 822	0.001	37.110	3.629	1 804	0.001	26.429	1.716	402	0.005	12.430	0.552
	502	1 622	0.001	26.650	3.567	1 609	0.001	24.733	1.477	346	0.005	58.720	0.682
	503	959	0.002	20.650	3.082	952	0.001	14.690	0.722	101	0.005	4.934	0.482
	504	267	0.005	27.100	3.853	267	0.002	15.700	2.060	53	0.005	5.350	0.381
3 = SE	501	0				0				0			
	502	0				0				0			
	503	1 247	0.001	39.390	3.970	1 232	0.001	19.750	0.175	1 230	0.005	15.350	0.725
	504	0				0				0			

Table 14-5 Sample statistics for the stratigraphy estimation

DEP	GEOLN	CU				ASCU				AU			
		Number	Minimum	Maximum	Mean	Number	Minimum	Maximum	Mean	Number	Minimum	Maximum	Mean
1 = Main	199	0				0				0			
	200	3	0.086	0.114	0.096	3	0.015	0.028	0.02	3	0.005	0.005	0.005
	201	80 616	0.001	58.653	0.307	78 859	0.001	23.937	0.122	52 042	0.005	69.5	0.061
	202	25 564	0.001	44.84	0.371	25 272	0.001	29.698	0.19	18 249	0.005	13.13	0.047
	203	20 747	0.001	51.36	1.051	20 414	0.001	41.345	0.337	13 768	0.005	75.093	0.187
	204	13 548	0.001	48.59	0.397	13 308	0.001	17.48	0.088	9 819	0.005	21.508	0.078
	205	41 421	0.001	34.7	0.186	40 425	0.001	20.7	0.043	34 827	0.005	16.547	0.036
	206	25 354	0.001	11.151	0.148	24 026	0.001	5.485	0.007	23 240	0.005	31.3	0.044
	207	6 122	0.001	10.87	0.061	3 962	0.001	0.874	0.003	3 963	0.005	11.034	0.034
	208	18 754	0.001	41.703	0.455	18 493	0.001	29	0.146	12 529	0.005	27.5	0.084
	210	5 175	0.001	25.33	0.755	5 024	0.001	15	0.087	4 589	0.005	9.227	0.13
	211	5 466	0.001	36.28	2.029	5 459	0.001	32.9	1.479	3 083	0.005	52.5	0.239
	212	0				0				0			
	213	0				0				0			

DEP	GEOLN	CU				ASCU				AU			
		Number	Minimum	Maximum	Mean	Number	Minimum	Maximum	Mean	Number	Minimum	Maximum	Mean
	300	1 846	0.001	6.72	0.188	1 579	0.001	3.67	0.071	1 075	0.005	1.148	0.021
	900	5 991	0.001	2.79	0.091	5 910	0.001	2.552	0.015	2 597	0.005	29.8	0.098
2 = NW	199	0				0				0			
	200	0				0				0			
	201	8 865	0.001	3.41	0.108	8 805	0.001	2.18	0.02	5 959	0.005	3.038	0.028
	202	18 464	0.001	33.22	0.234	18 123	0.001	27.39	0.116	13 186	0.005	9.193	0.05
	203	25 300	0.001	37.72	0.743	24 844	0.001	33.66	0.226	17 633	0.005	74.13	0.185
	204	20 364	0.001	18.51	0.054	19 864	0.001	3.45	0.01	17 457	0.005	11	0.021
	205	7 601	0.001	16.45	0.063	7 438	0.001	3.57	0.011	7 015	0.005	5.51	0.022
	206	8 934	0.001	18.323	0.057	7 437	0.001	1.94	0.008	7 242	0.005	3.66	0.022
	207	1 181	0.001	2.567	0.013	931	0.001	0.121	0.001	931	0.005	0.432	0.015
	208	18 673	0.001	43	0.439	18 412	0.001	31	0.197	12 968	0.005	58.72	0.095
	210	0				0				0			
2 = NW	211	5 979	0.001	28.1	0.695	5 920	0.001	27.25	0.338	4 395	0.005	14.1	0.063
	212	0				0				0			

DEP	GEOLN	CU				ASCU				AU			
		Number	Minimum	Maximum	Mean	Number	Minimum	Maximum	Mean	Number	Minimum	Maximum	Mean
	213	1 168	0.001	1.788	0.068	1 032	0.001	0.536	0.015	866	0.005	0.571	0.014
	300	0				0				0			
	900	5 633	0.001	5.9	0.141	5 593	0.001	5.1	0.032	3 364	0.005	3.149	0.055
3 = SE	199	14 927	0.001	50	0.062	14 655	0.001	33.3	0.018	14 407	0.005	4.601	0.017
	200	2 268	0.001	25.88	0.234	2 257	0.001	19.75	0.074	2 234	0.005	3.38	0.04
	201	49 786	0.001	39.39	0.289	49 492	0.001	21.7	0.017	48 934	0.005	20.9	0.057
	202	3 713	0.001	21.5	0.096	3 672	0.001	0.507	0.002	3 670	0.005	8.71	0.027
	203	6 896	0.001	30.4	0.333	6 755	0.001	2.57	0.007	6 755	0.005	17.567	0.068
	204	3 858	0.001	10.025	0.103	3 773	0.001	0.51	0.003	3 773	0.005	4.15	0.026
	205	4 915	0.001	17.62	0.094	4 561	0.001	0.143	0.002	4 559	0.005	6.33	0.023
	206	988	0.001	2.962	0.043	423	0.001	0.012	0.001	423	0.005	0.07	0.015
	207	425	0.001	0.643	0.013	13	0.001	0.001	0.001	13	0.005	0.01	0.006
	208	14 842	0.001	22.397	0.25	14 679	0.001	1.405	0.007	14 679	0.005	20.2	0.059
	210	0				0				0			
	211	0				0				0			

DEP	GEOLN	CU				ASCU				AU			
		Number	Minimum	Maximum	Mean	Number	Minimum	Maximum	Mean	Number	Minimum	Maximum	Mean
	212	880	0.001	6.17	0.52	880	0.001	6.12	0.231	880	0.005	2.79	0.061
	213	0				0				0			
	300	1 673	0.001	14.9	0.047	1 662	0.001	0.295	0.004	1 649	0.005	5.33	0.021
	900	2 183	0.001	3.924	0.077	2 147	0.001	1.112	0.011	2 114	0.005	6.53	0.033

14.9 **Specific gravity and Gangue Acid Consumption (GAC)**

In situ dry bulk density and GAC values were assigned to each block based on weathering and geological codes. At the beginning of 2012 the Kansanshi geology team began a routine programme of additional density measurements from the recently recovered diamond drilling core. Detailed procedures for the density determinations are outlined in KMP procedures (FQM, 2012).

Updated density assignments are presented in Table 14-6. Where units were missing bulk density and / or GAC data, values from similar units were assigned.

The weathering surfaces did not cover the extents of the block model in its entirety. Where blocks did not have a weathering code assigned from the wireframe, in a small section to the north-west of NW, elevation was used to differentiate between saprock and fresh material.

Table 14-6 Density and GAC values grouped by stratigraphy units and weathering zones

STRAT	WEATHN	WEATH	DENSITY	GAC	Assigned Density	Assigned GAC
UPS	1	FRSH	2.81	53		UPS 2012
	2	SAPR	2.66	53		UPS 2012
	3	SAP	1.66	36		UPS 2012
TM	1	FRSH	2.77	107		UM 2013
	2	SAPR	2.55	107	UM 2013	UM 2013
	3	SAP	1.90	61		UM 2013
UMC	1	FRSH	2.78	49		
	2	SAPR	2.10	37		
	3	SAP	1.80	38		
UM	1	FRSH	2.74	120		
	2	SAPR	2.55	70		
	3	SAP	1.90	49		
MMC	1	FRSH	2.80	62		
	2	SAPR	2.53	41		
	3	SAP	1.80	30	UMC 2013	
LCS	1	FRSH	2.78	95		

	2	SAPR	2.70	74		
	3	SAP	1.90	61	UM 2013	LCS 2012
LM	1	FRSH	2.75	97		
	2	SAPR	2.64	107		
	3	SAP	1.90	49	UM 2013	UM 2013
LPS	1	FRSH	2.80	95		UPS 2012
	2	SAPR	2.47	74		UPS 2012
	3	SAP	2.15	61		UPS 2012
LD	1	FRSH	2.81	97		LM 2013
	2	SAPR	2.64	107	LM 2013	LM 2013
	3	SAP	1.90	49	UM 2013	UM 2013
Lithology units						
	1	FRSH	2.76	49		UMC 2013
PHY	2	SAPR	2.10	37	UMC 2013	UMC 2013
	3	SAP	1.80	38	UMC 2013	UMC 2013
RES	1	FRSH	1.90	66	RES 2013	RES 2013
	2	SAPR	1.90	66	RES 2013	
	3	SAP	1.90	50		
DIOR	1	FRSH	2.85	50	DIO 2012	DIO 2012
	2	SAPR	2.50	50	DIO 2012	DIO 2012
	3	SAP	1.57	30	DIO 2012	DIO 2012
LAT	4	LAT	1.63	0	LAT 2012	LAT 2012

14.10 Block modelling

A 20 m x 20 m x 10 m (X x Y x Z) parent cell block model was built, sub-celled down to 1 m x 1 m x 1 m, to model the wireframed veins. It was then regularised to 20 m x 20 m x 10 m, with a field containing the proportion of vein (PV) informed by the wireframe. The dominant vein direction in the parent cell was used to inform the search ellipse. A 5 m x 5 m x 5 m SMU model was then created by regularising the parent cell model.

A 10 m x 10 m x 5 m SMU block model was created for the resource outside the wireframed vein mineralisation. This model was regularised to 40 m x 40 m x 5 m for grade estimation.

Dynamic Anisotropy was used for estimation of bedding dip and dip direction into the stratigraphy model in order to control the orientation of the search ellipse to ensure that it honoured the 'Dome' and dip of the stratigraphy units. This was undertaken in Datamine using the elevations from the strings generated from the stratigraphic wireframes supplied by the KMP mine geologists. Elevations of these strings were estimated into a block model (200 m x 200 m x 1 m) using Nearest Neighbour. These estimated elevations were used with the model centroid's X and Y's to generate a DTM. The dip and dip directions of the triangles in this DTM were to be estimated into the parent cells of the stratigraphy model.

The block models were flagged using stratigraphy surfaces, geology solids, vein wireframes, weathering surfaces, topography and depleted using pit outlines as at 31st May 2015.

Block model extents are presented in Table 14-7. Block model attributes are presented in Table 14-8.

Table 14-7 Block model local coordinate extents for final 10 m x 10 m x 5 m block model

Cell size (X)(m)	10
Cell Size (Y)(m)	10
Cell Size (Z)(m)	5
Origin (X)	630
Origin (Y)	7,500
Origin (Z)	800
Number of cells in X direction	844
Number of cells in Y direction	816
Number of cells in Z direction	140

Table 14-8 Block model attributes

Field Name	Description
IJK	Block position identifier
XC	Easting centroid
YC	Northing centroid
ZC	RL centroid
XINC	Dimension of block in X direction
YINC	Dimension of block in Y direction
ZINC	Dimension of block in Z direction
CU	Estimated Cu grade %
AU	Estimated Au grade g/t
ASCU	Estimated ASCu grade %
GEOLN	Number to define stratigraphic unit, vein, diorite, laterite
GEOL	Alpha equivalent of GEOLN
STRAT	Alpha equivalent of STRATN
STRATN	Stratigraphy number
WEATH	Alpha equivalent of WEATHN
WEATHN	Weathering: 1 = Fresh; 2 = saprock 3 = saprolite; 4 = laterite
OXMAT	Oxide category: - OXMAT=1=Ratio≤0.1 = Sulphide; OXMAT=2=Ratio 0.1-0.4 = Transitional; OXMAT=3=Ratio≥0.4, OXMAT=4 = Ratio >0.8 = highly leachable, high carbonate; OXMAT=5=Refractory (based on surface, REDOXN=3)
REDOX	Oxidation. Alpha equivalent of OXMAT
REDOXN	Oxidation Number: 1 = sulphide 2 = oxide/transitional; 3=refractory. Based on surfaces.
DENSITY	Density field provided by FQM
GAC	GAC provided by FQM
RATIO	Ratio of ASCU/TCU.

CLASS	Class field: 1=Measured, 2=Indicated, 3=Inferred, 9=Unclassified
PIT	Mined blocks: PIT=1=Mined in Main; PIT=2=Mined in NW
MINZON	0=Laterite; 1=remaining blocks
VEIN	Vein orientation for wireframe modelled veins: 1 = N-S; 2= E-W; 3=NW-SE and 4=NE-SW
DEP	Deposit field: 1 = Main; 2 = NW; 3 = SE

14.11 Grade estimate – VNGROUP1 – wireframed veins

Grade was estimated into the 20 m x 20 m x 10 m parent cells using data flagged as veins following the procedure described in Item 14.3. Grade was estimated using sample length weighted Inverse Distance Weighting to the power of 1. This approach was taken due to variably sized veins and variable length vein intercepts. Estimating variable width veins using composites can lead to bias in some cases. IDW allows for the use of length weighting in the estimate, an option not available when using Ordinary Kriging.

Clustering of the grade control data was managed by using separate searches for the parts of the deposit covered by grade control drilling, and parts covered by resource development and exploration drilling.

Since vein wireframes only represent volume, and not spatial location of veins, a localisation procedure was completed in Isatis. Each parent block had a proportion of vein, informed by the wireframe. This proportion was used alongside vein data to identify the 5 m x 5 m x5 m block within the parent that was most likely to have the vein material. The 5 m x 5 m x 5 m block was then assigned the grade of the parent cell.

14.12 Grade estimate – VNGROUP0 – stratigraphy

Grade was estimated into parent blocks using Uniform Conditioning in Isatis. The rotation of the search ellipse was informed by dip and dip direction estimated into the block model in the Dynamic Anisotropy process. Search ellipse ranges and sample search criteria are presented in Table 14-9.

Table 14-9 Sample search criteria and search ellipse ranges for stratigraphy

Pass	Range			Sample Search Criteria		
	U	V	W	Minimum	Maximum per sector	Sectors
1	100	100	7	18	10	4
2	200	200	15	18	10	4
3	200	200	15	7	14	1

Localised Uniform Conditioning of parent cells was completed to produce a 10 m x 10 m x 5 m block model, reflecting the SMU size.

14.13 Grade estimate – VNGROUP2 - veins not wireframed

There was vein material outside the vein wireframes and a broad estimate of this material was completed through the estimation of the proportion of a block that contains veins, and then assigning the mean grade for vein material in that stratigraphic unit. Proportions were estimated using IDW excluding vein data relating to vein wireframes.

To provide an indicative grade and tonnage for this vein material not yet wireframed, grade was not estimated. Rather, average Cu, AsCu and Au grades for each domain were used to assign grades into SMU blocks where the estimated proportion of vein exceeded 0.5. Where blocks also had a proportion associated with stratigraphic mineralisation, the grades were weighted. The amount of metal associated with this type of mineralisation comprises approximately 4% of the metal of the resource.

Table 14-10 Sample search criteria for vein estimation using non-grade control data

ESTZON		Search ellipse ranges			Minimum Samples	Maximum Samples	Max per drill hole
	Pass	X	Y	Z			
All ESTZON	1	15	80	30	9	18	18
	2	30	160	60			
	3	45	240	90	2	6	

Table 14-11 Search ellipse rotations for vein estimation using non-grade control data

ESTZON	Search ellipse rotations		
	z	x	y
1501,2501,3501	0	0	0
1502,2502	90	0	0
1503,2503,3503	315	0	0
1504,2504	45	0	0

Table 14-12 Search parameters for vein estimation using grade control data

ESTZON	Search ellipse ranges			Search ellipse rotations			Minimum Samples	Maximum Samples	Max per drill hole
	X	Y	Z	Z	X	Y			
1501,2501,3501	20	60	30	0	0	0	9	18	18
1502,2502				90					
1503,2503,3503				315					
1504,2504				45					

Note: One pass was completed to minimise clustering

Table 14-13 Average grades used for VNGROUP2 material

DEP	GEOLN	AVGRADE CONTROLU	AVGASCU	AVGAU
Main	201	1.32	0.44	0.44
	202	3.59	1.65	0.44
	203	3.02	0.29	0.99
	204	2.64	0.39	0.39
	205	1.76	0.30	0.25
	206	0.80	0.05	0.19
	207	0.62	0.03	0.12
	208	4.10	0.32	0.79
	210	1.76	0.30	0.24
NW	201	0.09	0.01	0.06
	202	1.07	0.73	0.22
	203	1.26	0.14	0.30
	204	0.55	0.03	0.13
	205	0.87	0.05	0.17
	206	1.15	0.04	0.18
	208	2.66	1.10	0.47

	211	1.30	0.93	0.13
SE Dome	199	0.50	0.21	0.08
	200	1.66	0.48	0.21
	201	2.13	0.07	0.40
	203	2.28	0.04	0.19
	204	1.27	0.03	0.30
	205	1.03	0.01	0.18

14.14 Block model validation

The stratigraphy model and the localised vein estimate (5 m x 5 m x 5 m) was combined with the localised stratigraphy model (10 m x 10 m x 10 m) to create the final block model.

Validation comprised:

- comparing wireframed vein volumes with block model vein volumes
- viewing cross sections of sample grade data versus block model grade estimates
- comparing mean grades of data with mean grades of the block model
- swath plots along northings, eastings and RL

The wireframes and block model volumes compared well, and were within 1%, a difference which can be attributed to a slight loss of resolution at block edges. Examples of cross sections are presented in Figure 14-5 and Figure 14-6. Visual validation suggests the grade tenor of the input data is reflected well in the block model. For veins, when input and model grades for veins are compared globally, Cu and Au validate reasonably well for the vein data in the vein model, Cu validates to within 5% and Au validates to within 10%.

Average AsCu grades were 30% lower for the vein model than the data. The global differences for AsCu can be attributed to the fact that AsCu grades change quite dramatically between oxidation units, with high grades in oxide and lower grades in refractory and sulphide. The proportion of the data represented by oxide was 56% but it only represents 21% of the vein resource in the block model. The proportion of the dataset represented by sulphide was 47% but it represents 78% of the vein resource in the block model. Therefore, the lower grades in the sulphide are biasing the global block model grades towards lower grades; while the high grades in the oxide are biasing the data statistics towards higher grades in the data. When reviewing diamond development drilling alone, the proportions of each oxidation unit drilled were more comparable to the block model. However, the number of grade control data far exceeds that of resource development, and therefore the statistics favour the grade control data.

When AsCu is compared by oxidation unit, sulphide was down 9%, oxide was up 4% and refractory was up 7% in the model compared to the data. In summary, AsCu validates well in the block model when compared with the input data.

For the stratigraphic model, grades in the model were generally lower than grades in the data. However, this was also a function of extrapolation of lower grades at the edges of the deposit that have poorer sample support. A string was digitised to limit the model and data for validation purposes and these validated reasonably well.

Swath plots were perhaps the most reliable way to validate the model on a semi-local scale. Swaths were produced for the veins and major stratigraphic units, and overall validated quite well where there was sufficient data informing the block estimates. For veins, there may be some benefit in assigning the mean grade for veins to blocks that have low quality grade estimates. It was not recommended that this strategy be applied to stratigraphy without accounting for spatial trends away from the domes.

Figure 14-5 Vein model and drill data, vertical sections showing North West Pit (top), Main Pit (middle) and South East Dome (bottom)

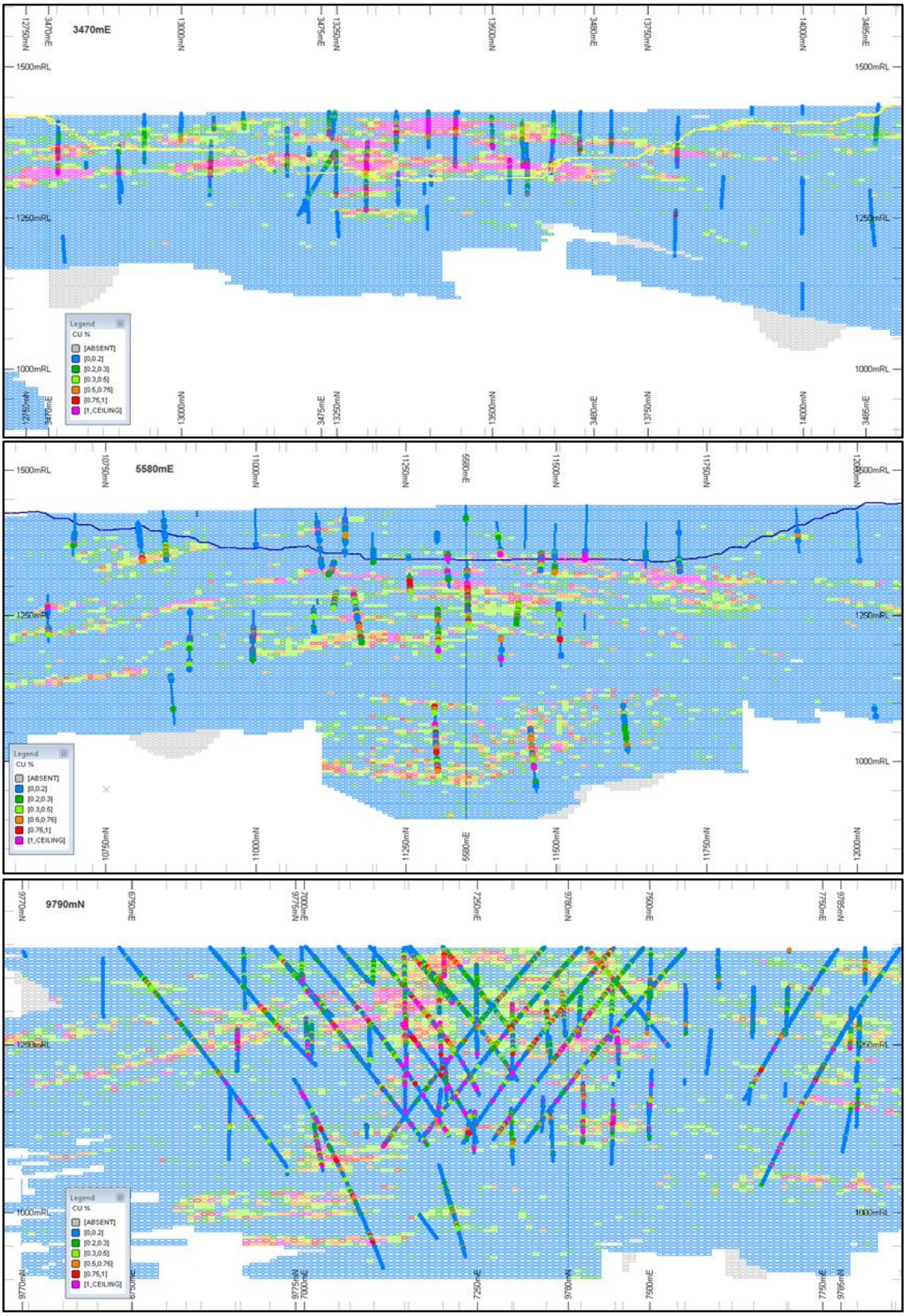


Figure 14-6 Vein LUC model SMU (coloured) showing parent cells (grey) and drill hole data, vertical sections showing North West Pit (top), Main Pit (middle) and South East Dome (bottom)

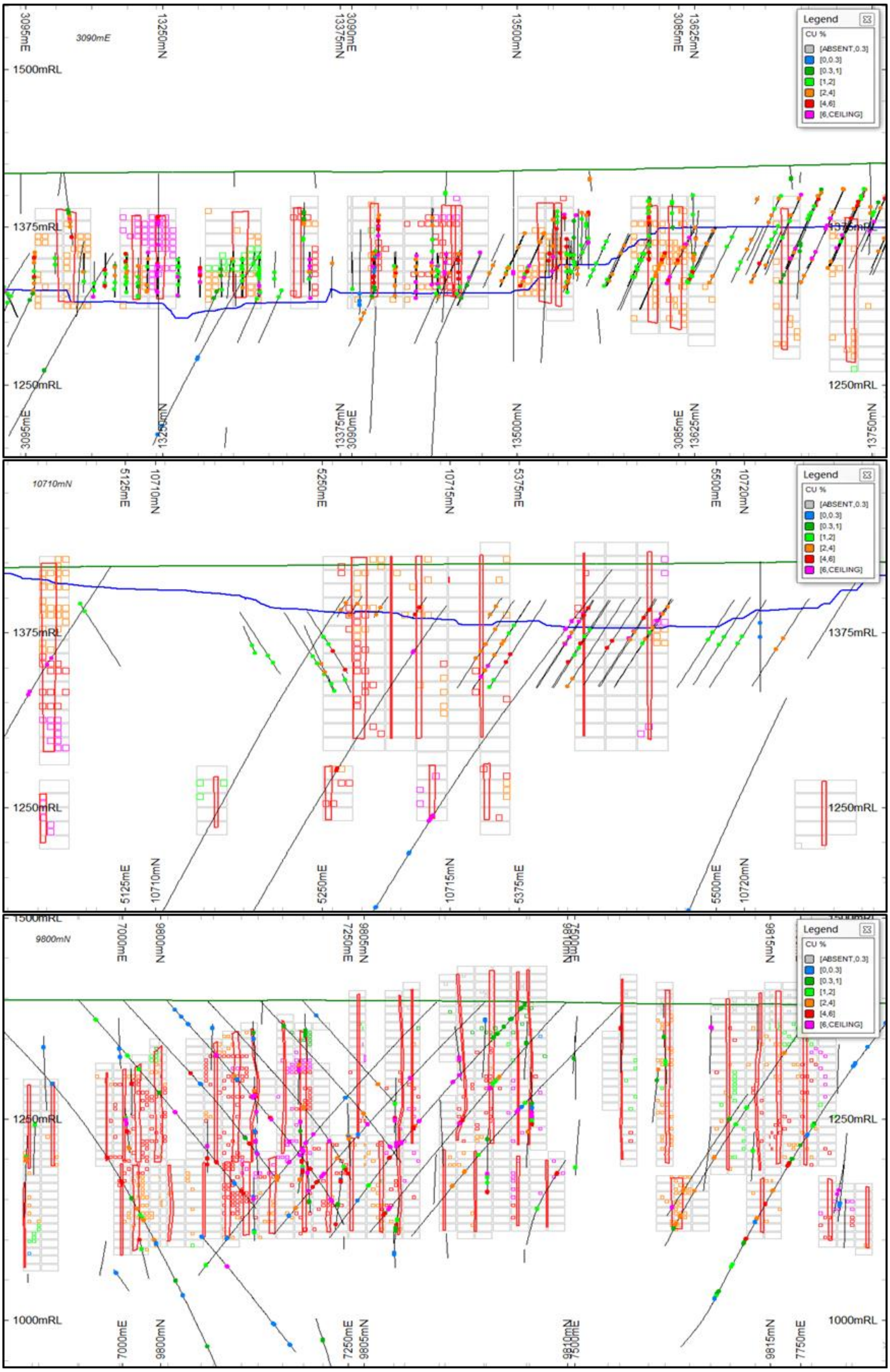


Figure 14-7 Northing swath plot showing vein model for Cu in Main Pit

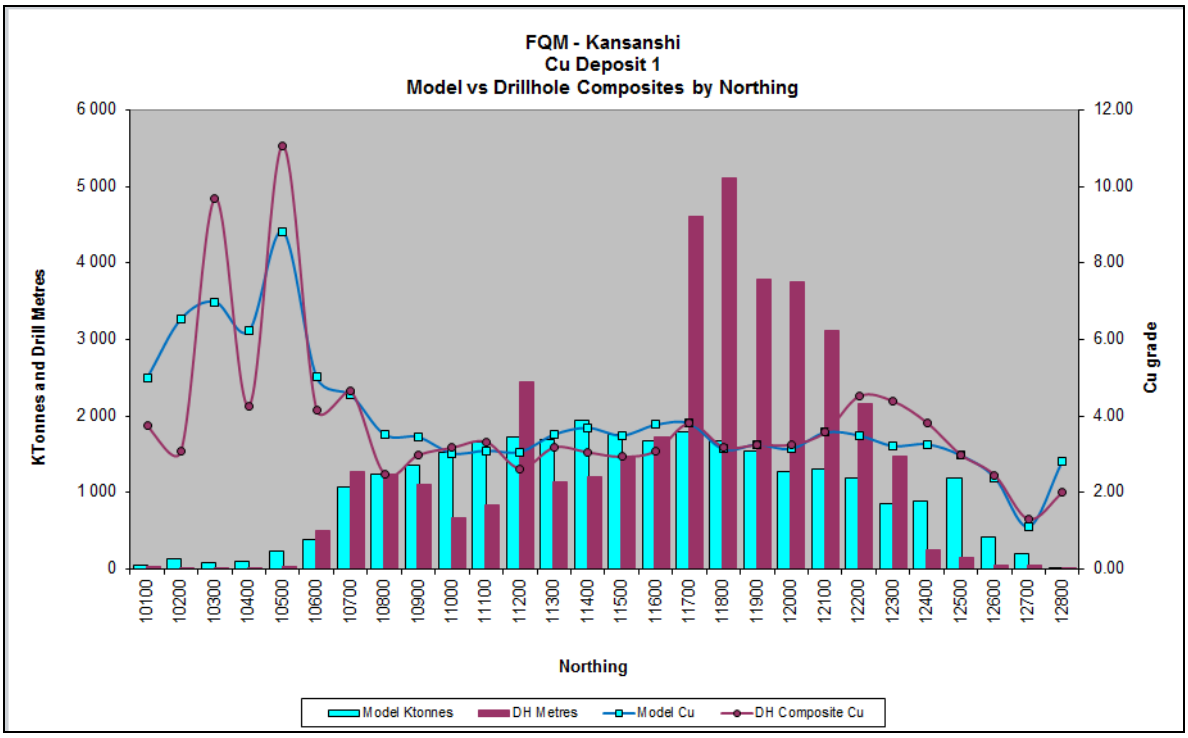


Figure 14-8 Easting swath plot showing vein model for Cu in North West Pit

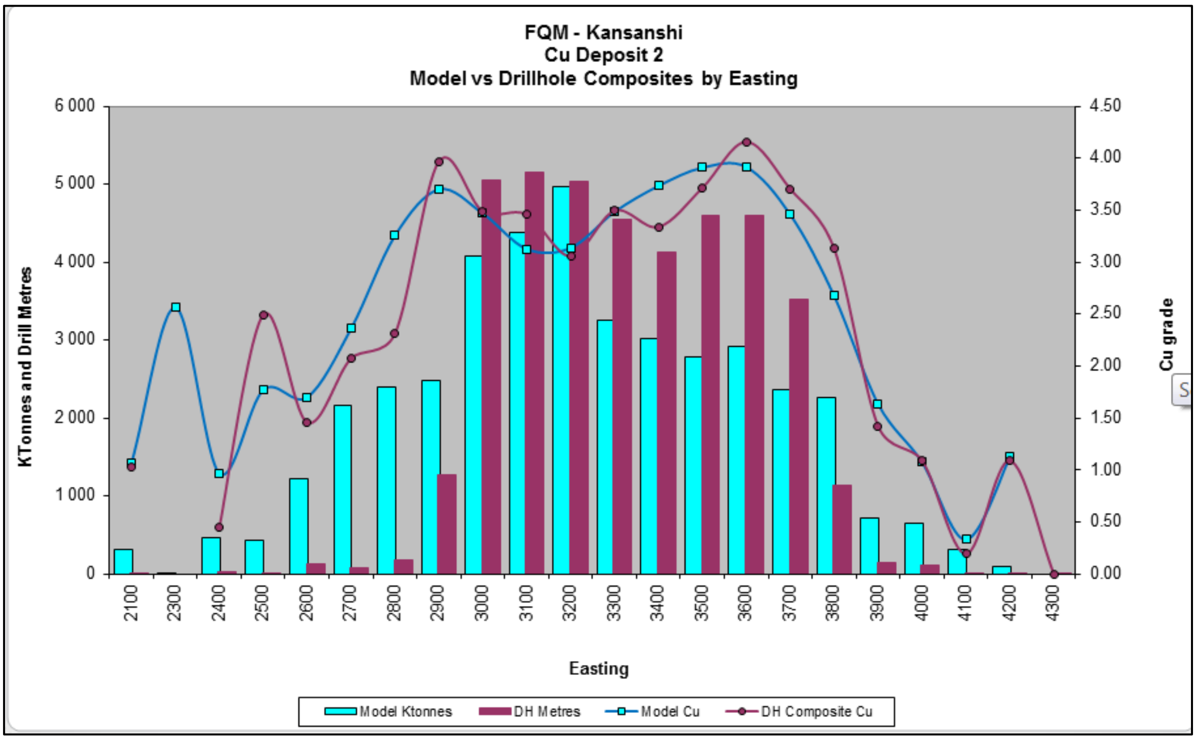


Figure 14-9 Easting swath plot showing UMC for Cu (all deposits)

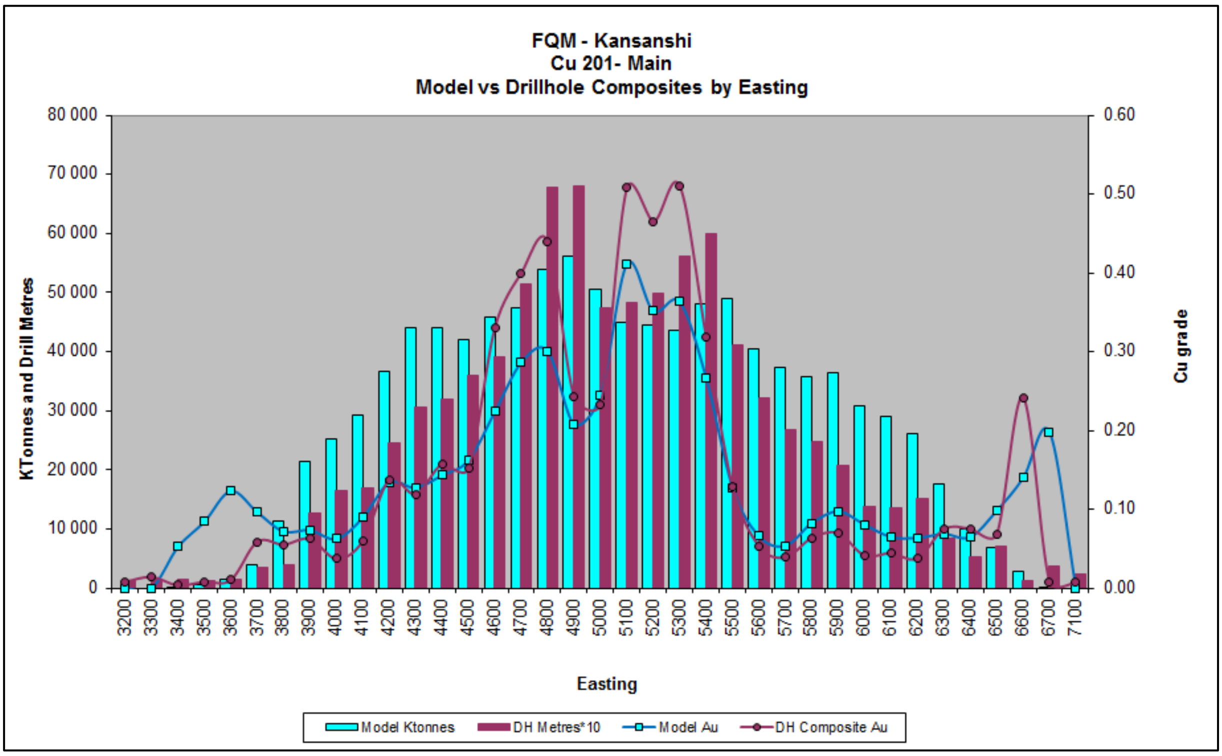
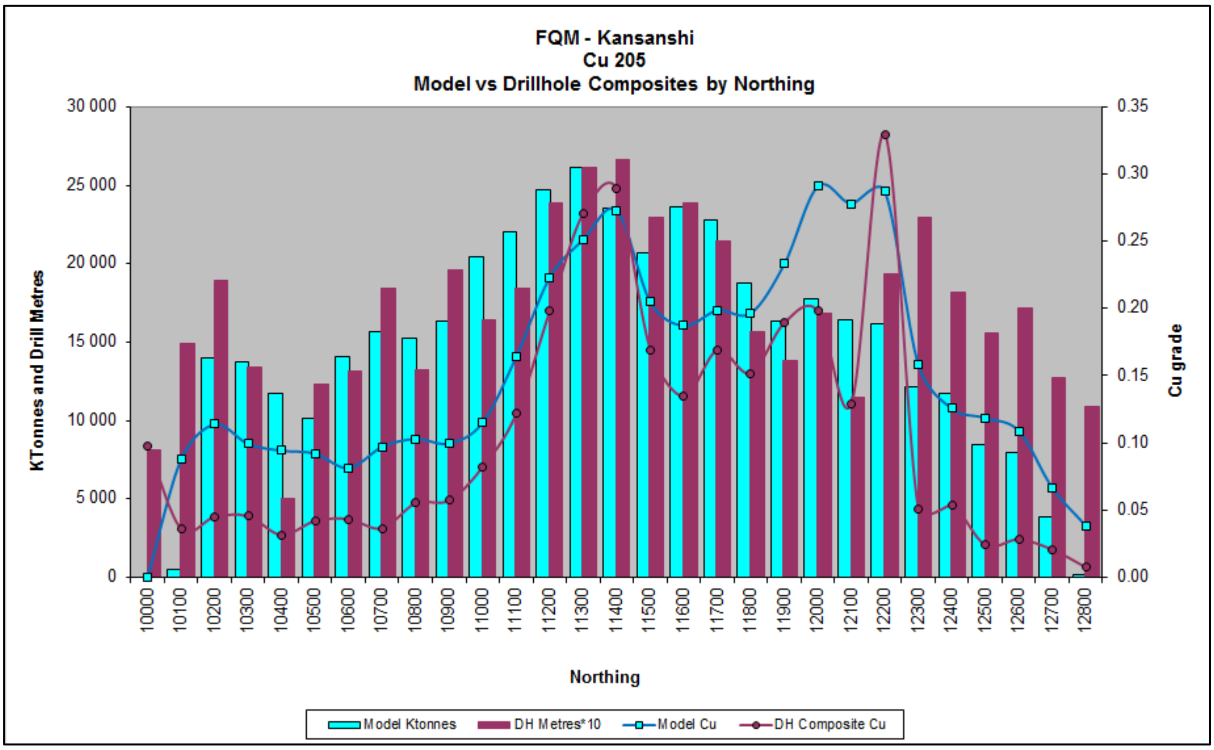


Figure 14-10 Easting swath plot showing LM for Cu (all deposits)



14.15 Resource classification

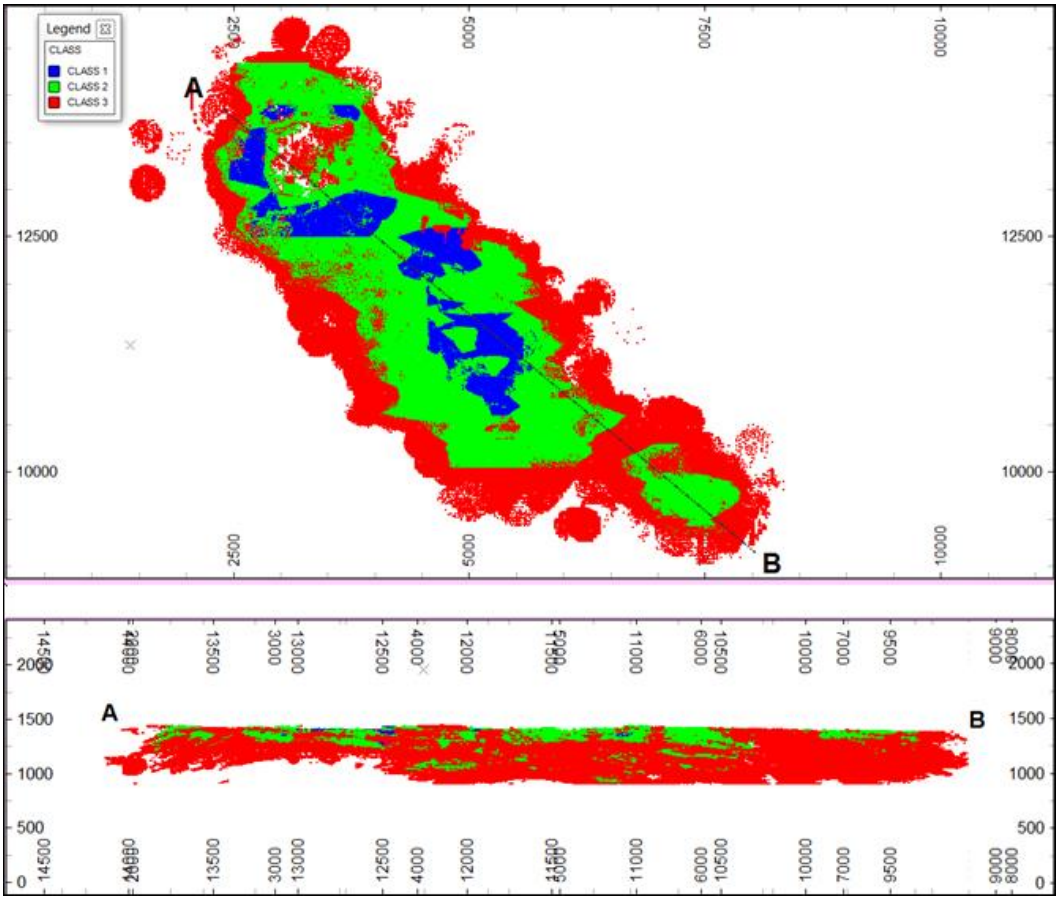
The Mineral Resource estimate was classified as Measured, Indicated and Inferred (Figure 14-11) in accordance with the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (the CIM Guidelines, 2014), and also comply with the guidelines of the JORC Code (JORC, 2012). Classification was based on a number of criteria including assessment of the reliability of the geological model, sampling, survey control, bulk density data, drilling grid, and Kriging statistics. Specifically, the following criteria were considered by the QP, David Gray and CSA:

- Adequate validation of tenement title, drilling, sampling, and geological process completed during site visits by David Gray and Malcolm Titley between 2013 and 2015.
- Adequate geological evidence for continuity of mineralization at the cut-off grade used in the estimation of the mineral resource.
- Adequate analytical evidence of copper, acid soluble copper and gold mineralisation.
- Adequate QA/QC controls in place to validate the copper and gold grades.
- Adequate diamond core sampling to determine the dry in situ bulk density in order to estimate the tonnage of mineralisation.
- Review of the Ordinary Kriging slope of regression as an indicator of relative confidence of the grade estimate.

The Kriging 'confidence' measured by the slope of regression was combined with geological confidence and sample spacing as a guide to determining classification boundaries:

- For Measured, wireframes were constructed that broadly delineate blocks that were estimated in the first search pass and had a slope>0.7. Veins that fell within this wireframe were also classified as Measured.
- For Indicated, wireframes were constructed that broadly delineate blocks that were estimated in the first or second search pass and had a slope >0.5. Veins that fell within this wireframe were also classified as Indicated.
- Inferred material was assigned to the remaining block estimates. Diorite was also classified as Inferred.

Figure 14-11 Plan view (top) and cross section view (bottom) of the Kansanshi block model, coloured by CLASS, where Blue is Measured, Green is Indicated and Red is Inferred



14.16 Mineral Resource reporting

The updated Mineral Resource estimate for Kansanshi is presented Table 14-14. The Mineral Resource is reported at a 0.2% total copper cut-off grade as per the latest reserve modifying factors and financial evaluations detailed in this report. Laterite mineralisation was not reported due to risk associated with logging input data. Mineral Resources are inclusinve of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 14-14 Kansanshi Operations Mineral Resource statement, at May 2015

First Quantum Minerals Limited - Kansanshi Copper Gold Mine Dec 2013 Mineral Resource Estimate depleted to 31st May 2015 Reported at a cut-off grade of 0.2% Tcu						
Classification	**Tonnage (Mt)**	**TCu%**	**Kt Cu**	**ASCu%**	**Au g/t**	**Density**
Measured Resource	105	0.71	738	0.21	0.12	2.44
Indicated Resource	735	0.72	5,313	0.15	0.12	2.61
Total Measured & Indicated	**840**	**0.72**	**6,051**	**0.16**	**0.12**	**2.59**
Inferred Resource	669	0.60	3,993	0.04	0.10	2.75

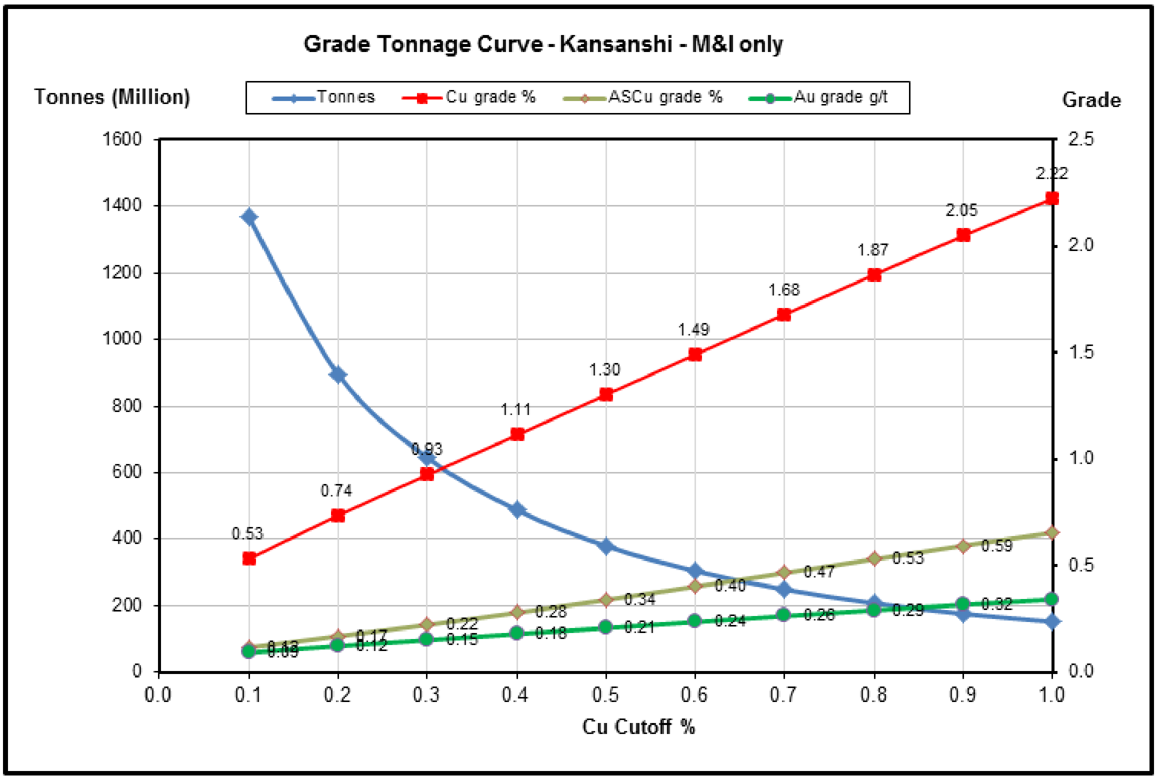
To the best knowledge of the QP, David Gray, the stated Mineral Resource is not materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant issues that prevent this resource from reasonable prospects for economic extraction. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Grade and tonnage tabulations of the Measured and Indicated material only are presented in Figure 14-12.

Figure 14-12 Kansanshi grade and tonnage tabulation for Measured and Indicated Mineral Resources.

Grade and Tonnage Tabulation - Kansanshi - M&I Resource

Cu % Cut-off	Tonnage (Million)	Cu grade %	Cu Metal KTonnes	ASCu grade %	ASCu Metal KTonnes	Au grade g/t	Au Metal Koz	In-situ Dry Density
1.0	152	2.22	3 372	0.66	994	0.34	1 665	2.56
0.9	175	2.05	3 598	0.59	1 044	0.32	1 787	2.56
0.8	207	1.87	3 865	0.53	1 098	0.29	1 934	2.56
0.7	248	1.68	4 174	0.47	1 157	0.26	2 104	2.56
0.6	304	1.49	4 532	0.40	1 220	0.24	2 303	2.56
0.5	379	1.30	4 946	0.34	1 287	0.21	2 531	2.56
0.4	488	1.11	5 430	0.28	1 359	0.18	2 805	2.56
0.3	645	0.93	5 974	0.22	1 434	0.15	3 125	2.56
0.2	895	0.74	6 590	0.17	1 510	0.12	3 518	2.56
0.1	1 367	0.53	7 270	0.12	1 585	0.09	4 041	2.56



14.17 Comparison with previous Mineral Resource estimate

Several risks and recommendations were highlighted in the December 2012 Technical Report (Titley, CSA, 2012), and to address these, changes were made to the geological model and estimation methodology used. These include:

- The vein model had a significant amount of dilution in it, and efforts were made by FQM to wireframe veins using a more rigorous, and mathematical approach, restricted only to stratigraphic units that have well developed veining (MMC, UMC).
- Uniform conditioning was used as a recoverable estimation technique to estimate the mineralisation in the stratigraphy. It is believed that this has provided for a more reliable estimate of the mineralisation in the stratigraphy, which has moved away from the probability approach which was required in 2012.
- Wireframes representing oxidation boundaries produced previously were subjective and did not adequately constrain oxide and transitional material. The 2013 model has used the ratio of ASCu/TCu to estimate the oxidation characteristics of blocks. This is the result of internal analysis work completed by FQM and reviewed by CSA.
- The changes in the geological model had a significant impact on the vein component of the model. The contained metal in the vein model decreased from 2,980 kt Cu to 2,110 kt Cu. Tonnage decreased from 242 Mt to 62 Mt and grade increased from 1.23% Cu to 3.43% Cu. There was also an increase in Au from 0.20 g/t to 0.59g/t.
- There is a 19% drop in tonnes and a 6% increase in TCu grade from the December 2012 to the May 2015 Mineral Resources (Table 14-15).
- 10% of the reduced tonnages and copper metal is due to mining since December 2012. Accordingly, changes to this resource estimate have resulted in a 9% drop in tonnes, a 6% increase in TCu grade and a 5% loss in total copper metal. This is mainly due to the improved vein and stratigraphy estimation methods together with the confidence increases and extensions from South East dome drilling.

Table 14-15 Comparison between the December 2012 Mineral Resource estimate and the May 2015 Mineral Resource estimate using a 0.3% TCu cutoff.

	Tonnage (Mt)	TCu%	Kt Cu	ASCu%
Measured Resource	-12%	-22%	-31%	-45%
Indicated Resource	-20%	11%	-12%	10%
Total Measured & Indicated	**-19%**	**6%**	**-15%**	**-2%**
Inferred Resource	17%	12%	31%	50%

The QP, David Gray deems this Mineral Resource estimate update robust and representative of its input data, the prevailing geology and detail knowledge gained from mining activities over the last two years. Model comparisons are justified and acceptable within the context of confidences associated the previous Mineral Resource classifications.

ITEM 15 MINERAL RESERVE ESTIMATE

Detailed technical information provided under this item relates specifically to the Mineral Reserve estimate completed for this Technical Report and based on the Mineral Resource models and estimate as reported in Item 14. This new information has been authored by Michael Lawlor (QP).

15.1 Introduction

As part of the estimation process, pit optimisation aspects and detailed pit designs were completed by FQM personnel overseen and supervised by Michael Lawlor (QP) of FQM. 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 Resources only. The Measured and Indicated Mineral Resource estimates as listed in Table 14-14 are reported inclusive of the Mineral Reserves.

15.2 Methodology

The conversion of the Mineral Resource estimate to a Mineral Reserve estimate followed a conventional approach, commencing with open pit optimisation techniques incorporating economic parameters and other “modifying” factors. The optimisations were carried out in 2013/2014 following a review of Project operating and metal costs (ie, transport costs and refining charges (TCRCs) based on 2012 actual data. At the time, the metal cost inputs to the optimisation considered concentrate treatment and refining only. The impact of replacement smelting costs and anode sales was considered subsequently in an optimisation sensitivity analysis and in the cashflow model supporting the Mineral Reserve estimate (Item 22).

The ultimate (optimal) shells were used to create practical and detailed open pit designs accounting for the siting of in-pit crushers, conveyor ramps, batters, berms and haul roads. These designs were commenced in 2013 and revised several times during 2014 to account for changes to the siting of in-pit crushers.

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

Versions of the life of mine (LOM) schedule were produced in 2014 to reflect changes in the optimisation of plant feed types and the commencement of S3 processing. A mining inventory and production schedule were produced for Q1 2015 Annual Information Form reporting (as at end of December 2014) and this schedule has now been depleted/adjusted to reflect the Mineral Reserve estimate, as at 31st May 2015.

15.3 Mine planning models

A regularised block model to 10 m x 10 m x 5 m was provided for pit optimisation and mine planning work. For the sake of time and efficiency in dealing with an otherwise large model, the regularised model was reblocked to 30 m x 30 m x 10 m, and without compromising the definition of the original model.

15.4 Pit optimisation

15.4.1 Optimisation methodology

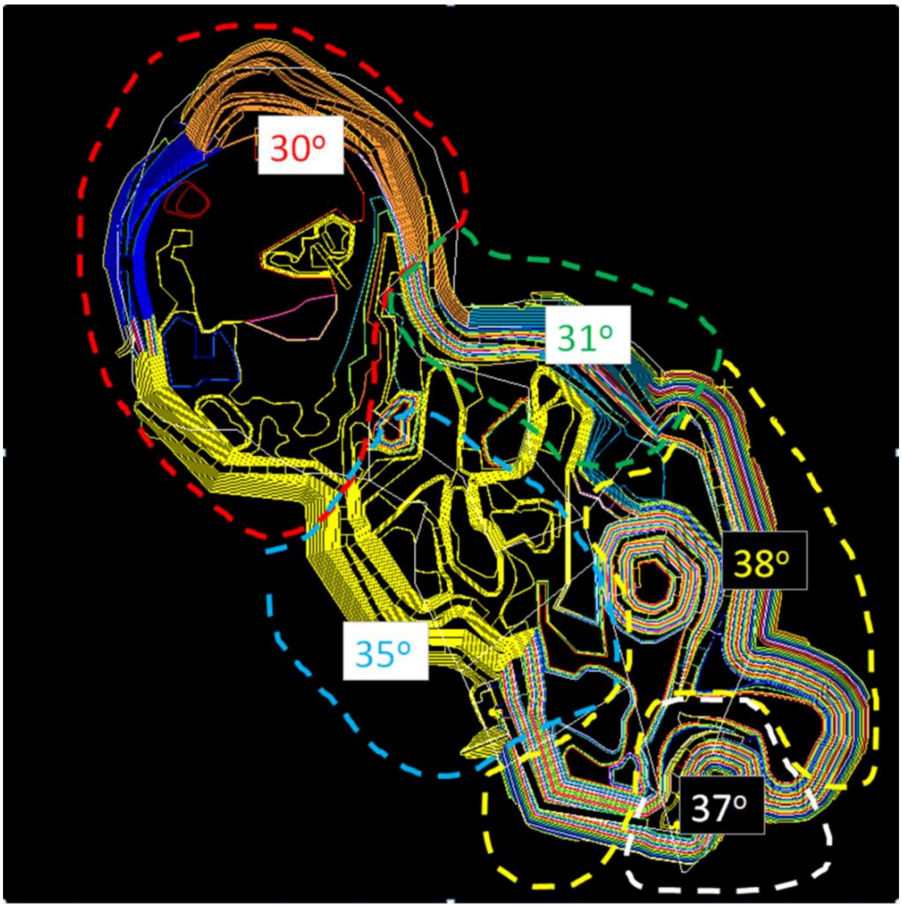
Conventional Whittle Four-X software was used to determine optimal pit shells for each of the various deposits. Optimisations were completed on a maximum net return (NR) basis, and with recoveries to copper metal in concentrate or as cathode determined from variable and fixed recovery relationships.

15.4.2 Optimisation input parameters

Pit slope design criteria

Pit optimisation input included overall slope design angles as shown in Figure 15-1. These parameters are a simplification, for the purposes of optimisation, from a comprehensive listing of slope design parameters the geotechnical engineering basis for which is outlined in Item 16.

Figure 15-1 Overall pit slope angles for optimisation input



Metal prices

The optimisation inputs for metal prices were as follows:

- Copper = US$3.00/lb (US$6,615/t)
- Gold = US$1,200/oz

Metal recoveries

Variable input process recovery projections were based on a set of equations derived by KMP process engineers (Table 15-1). These equations have been updated from those reported in the 2012 Technical Report (Journet and Cameron, DumpSolver, 2012). The recoveries for copper to cathode have changed and recoveries of copper in the mixed and sulphide feed are now capped to certain maximum numbers. The adopted copper concentrate grade is now 26.5%.

Table 15-1 Process recovery equations

DESCRIPTION	2014
Acid insoluble Cu grade	IF(cu.G>0,cu.G-ascu.G,0)
Cu recovery to Oxide Conc (not in %)	IF(ascu.G>0,(((0.035*ascu.G)+(0.40*aicu))/cu.G),0)
Cu recovery to Cathode (not in %)	IF(ascu.G>0,(((0.92*ascu.G*(1-0.035))+(0.45*aicu*(1-0.4)))/((ascu.G*(1-0.035))+(aicu*(1-0.4))))*0.98,0)
Cu in Tailing from Mix Plant (in %)	IF(cu.G>0,MIN(cu.G,((0.0688*cu.G)+(0.8*ascu.G))),0)
Cu Recovery to Mix Conc (in %)	IF(cu.G>0,IF((cu.G-T3)/(26.5-T3)*26.5/cu.G<0,0,MIN(0.81,(cu.G-T3)/(26.5-T3)*26.5/cu.G)),0)
Cu in Tailing from Sul Plant (in %)	IF(cu.G>0,MIN(cu.G,((0.0329*cu.G)+(1.5132*ascu.G))),0)
Cu Recovery to Sul Conc (in %)	IF(cu.G>0,IF((cu.G-T4)/(26.5-T4)*26.5/cu.G<0,0,MIN(0.92,(cu.G-T4)/(26.5-T4)*26.5/cu.G)),0)
Recovery Gold in Oxide Gravity	0.753
Recovery Gold in Mix Gravity	0.434
Recovery Gold in Sulphide Gravity	0.21
Recovery Gold through Oxide Conc	0.247
Recovery Gold through Mix Conc	0.566
Recovery Gold through Sulphide Conc	0.79

Operating costs

Since the Project will be mill constrained, the process operating costs are the sum of the fixed and variable costs. These costs have been updated from those reported in the 2012 Technical Report (Journet and Cameron, 2012), and are now as follows:

- operating costs = $8.44/t for mixed leach feed, $7.05/t for mixed float feed, and $4.89/t for sulphide float feed
- fixed costs (equivalent G&A costs in variable terms) = $5.10/t for mixed leach feed, $3.77/t for mixed float feed, and $4.00/t for sulphide float feed
- total operating costs = $13.54/t for mixed leach feed, $10.82/t for mixed float feed, and $8.88/t for sulphide float feed

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

Variable mining costs comprising drill, blast, load and haul costs, on a bench by bench basis, were determined from haulage simulations and from reference to FQMO actual costs. Item 21 outlines the derivation of the following incremental relationship adopted for ore and waste mining costs in the pit optimisation process:

- ore mining ($/t) = -0.0019 x RL + 3.95
- waste mining ($/t) = -0.0047 x RL + 7.51

Metal costs

In addition to 6% (gross) royalties, and at the time of the optimisation in 2014, metal costs for the Kansanshi product streams comprised:

- cathode metal production costs (SX and EW consumables)
- EW power costs
- concentrate treatment charges and refining charges

Item 21 provides an explanation on the derivation of the following metal costs used for pit optimisation:

- total cathode copper metal costs = $0.36/lb Cu
- total copper in concentrate metal costs = $0.36/lb Cu
- average float/leach feed metal costs = $0.36/lb Cu
- total gold in copper concentrate metal costs = $556.35/oz for float/leach feed and $418.03/oz for sulphide float feed (based on overall average head grades and copper recovery values)
- total gold in gravity circuit metal costs = $92.00/oz

Item 21 also provides an update on the replacement metal costs applicable for smelting charges and anode sales, and inclusive of 9% (rather than 6%) royalties, which are:

- total cathode copper metal costs = $0.45/lb Cu
- total copper in anode metal costs = $0.59/lb Cu
- average float/leach feed metal costs = $0.54/lb Cu
- total gold in copper concentrate metal costs = $247.20/oz for float/leach feed and for sulphide float feed (based on overall average head grades and copper recovery values)
- total gold in gravity circuit metal costs = $128.00/oz

Net return (NR)

Table 15-2 lists the notional NR values for the sale of concentrate and cathode, based on overall average recoveries and model grades. Table 15-3 lists the comparable NR values based on replacement smelting charges and anode sales.

To avoid confusion, the NR (ie, Net return = recovery * (revenue – metal costs)) must be expressed in units of metal grade. Since the metal grades are in % terms, this is $/10kg. In other words, the $/lb costs must be multiplied by 2,204.62 and divided by 100.

Mining dilution and recovery factors

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

Item 16.3 provides further commentary on these factors, in the light of information gleaned from operational tracking and reconciliation.

Table 15-2 Kansanshi Net Return values, cathode and concentrate TCRCs

	Units	Float/leach		Float
		Leach	Mixed	Sulphide
Processing Parameters:				
Smelter recovery	%			
Overall average copper recovery	%	80.0	74.0	89.0
Average gold to concentrate recovery	%	15.9	38.3	50.1
Average god to dore (gravity) recovery	%	48.6	29.4	13.3
Overall average gold recovery	%	64.5	67.7	63.4
Average Grades:				
Cu	%	0.78	0.72	0.61
Au	g/t	0.15	0.13	0.12
Price less Metal Costs:				
Cu Metal Price	$/lb	3.00	3.00	3.00
Cu Metal Cost	$/lb	0.36	0.36	0.36
Cu Net Return	$/lb	2.64	2.64	2.64
Cu Net Return (recovered)	**$/lb**	**2.11**	**1.95**	**2.35**
Au Metal Price	$/oz	1,200.00	1,200.00	1,200.00
Au Metal Cost in dore	$/oz	92.00	92.00	92.00
Au Metal Cost in concentrate	$/oz	305.00	556.34	481.03
Au Net Return	$/oz	2,003.00	1,751.66	1,826.97
Au Net Return	$/lb	29,209.75	25,544.44	26,642.67
Au Net Return (recovered)	$/lb	18,840.29	17,293.59	16,891.45
CuEq Net Return (recovered)	**$/lb**	**0.29**	**0.29**	**0.24**
Total Net Return (recovered)	**$/lb**	**2.40**	**2.24**	**2.58**
Total Net Return (recovered)	**$/10kg**	**52.98**	**49.30**	**56.93**

Table 15-3 Kansanshi Net Return values, cathode and anode TCRCs

	Units	Float/leach		Float
		Leach	Mixed	Sulphide
Processing Parameters:				
Smelter recovery	%	95.7	95.7	95.7
Overall average copper recovery	%	80.0	74.0	89.0
Average gold to concentrate recovery	%	15.9	38.3	50.1
Average god to dore (gravity) recovery	%	48.6	29.4	13.3
Overall average gold recovery	%	64.5	67.7	63.4
Average Grades:				
Cu	%	0.78	0.72	0.61
Au	g/t	0.15	0.13	0.12
Price less Metal Costs:				
Cu Metal Price	$/lb	3.00	3.00	3.00
Cu Metal Cost	$/lb	0.54	0.59	0.59
Cu Net Return	$/lb	2.46	2.41	2.41
Cu Net Return (recovered)	**$/lb**	**1.97**	**1.78**	**2.14**
Au Metal Price	$/oz	1,200.00	1,200.00	1,200.00
Au Metal Cost in dore	$/oz	128.00	128.00	128.00
Au Metal Cost in concentrate	$/oz	247.20	247.20	247.20
Au Net Return	$/oz	2,024.80	2,024.80	2,024.80
Au Net Return	$/lb	29,527.66	29,527.66	29,527.66
Au Net Return (recovered)	$/lb	19,045.34	19,990.22	18,720.54
CuEq Net Return (recovered)	**$/lb**	**0.30**	**0.33**	**0.26**
Total Net Return (recovered)	**$/lb**	**2.27**	**2.11**	**2.40**
Total Net Return (recovered)	**$/10kg**	**49.97**	**46.52**	**52.98**

15.4.3 **Marginal cut-off grades**

Whittle uses the following simplified formula to calculate the marginal cut-off grade as listed in Table 15-4 for production of cathode and concentrate, and Table 15-6 for cathode and anode. The tabled grades are indicative overall average cut-off grades; the actual cut-off grade for each block varies due to the variable process recovery relationships.

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

Table 15-4 Kansanshi marginal cut-off grades, cathode and concentrate TCRCs, 6% royalty

	Units	Float/leach		Float
		Leach	Mixed	Sulphide
Marginal Cut-Off Grade:				
PROCOST	$/t ore	13.54	10.82	8.88
MINDIL factor		1.05	1.05	1.05
TOTAL NET RETURN	$/10kg	52.98	49.30	56.93
C/O GRADE	**%Cu**	**0.27**	**0.23**	**0.16**

Table 15-5 Kansanshi marginal cut-off grades, cathode and anode TCRCs, 9% royalty

	Units	Float/leach		Float
		Leach	Mixed	Sulphide
Marginal Cut-Off Grade:				
PROCOST	$/t ore	13.54	10.82	8.88
MINDIL factor		1.05	1.05	1.05
TOTAL NET RETURN	$/10kg	49.97	46.52	52.98
C/O GRADE	**%Cu**	**0.28**	**0.24**	**0.18**

15.4.4 **Optimisation results**

Figure 15-2 shows the graphical result of pit optimisation. The optimal shell was selected on a maximum net return (undiscounted) basis. Table 15-6 lists the complete inventory of shell sizes and corresponding cashflows.

15.4.5 **Optimisation sensitivity analyses**

Optimisation sensitivity analyses were completed to test the impact of varying copper price, overall copper recovery, mining dilution, royalties, mining and processing costs. The analyses also show that the impact of increasing the royalty from 6% to 9%, ie to the now prevailing level, results in an approximate 7% reduction in undiscounted project value. Although the project value is reduced, the impact on the marginal cut-off grade is minimal, as shown in Table 15-9 (compare with the grades in Table 15-4). On this basis, there is no need to revise the optimisations, pit designs and production schedules to account for the increase. The impact could be addressed as a cashflow sensitive variable.

Table 15-7 shows the results of the analysis and confirms that selling price is the most sensitive variable (sensibly, since this is related to net return). Mining dilution is also a relatively sensitive variable, whilst mining and processing (including G&A) costs are less so.

Figure 15-2 Kansanshi optimisation results

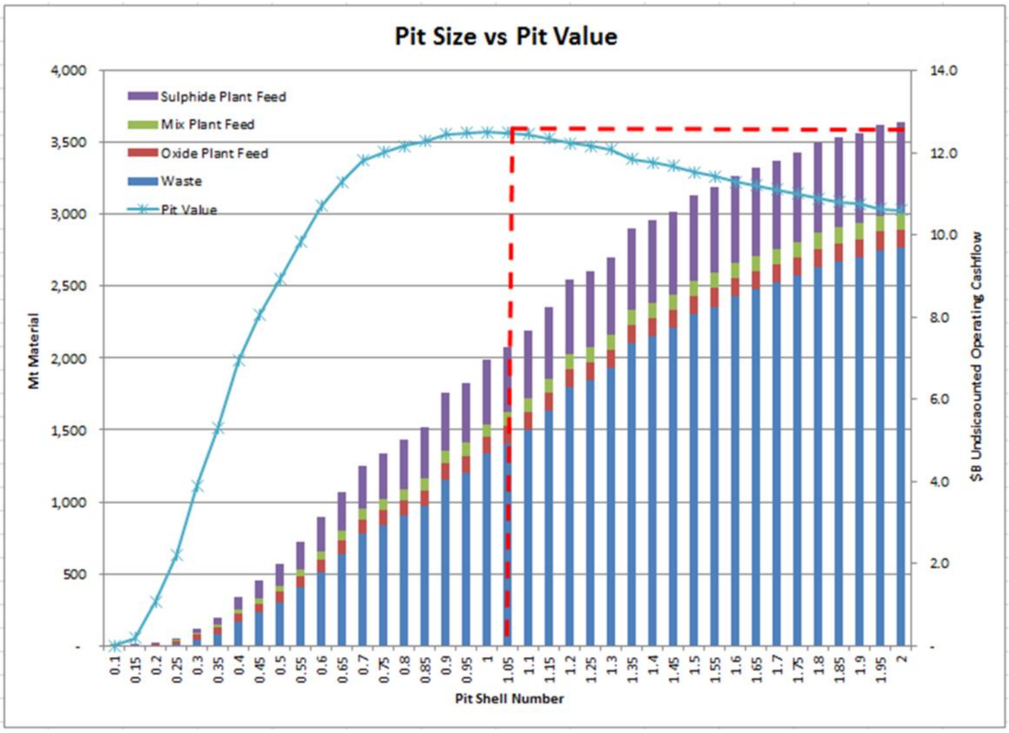


Table 15-6 Pit optimisation shell sizes and cashflow

Pit	Material	Total Ore	Strip	Plant Feed		Material Processed			Metal Recovered				Recovery		Pit
Number	Mined	Processed	Ratio	Grade	Insitu Cu	Oxide Plant	Mix Plant	Sulphide Plant	Cu in Conc	Cathode	Au in Conc	Au in Dore	Cu & ASCu	Au	Value
	Mt	Mt	Waste : Ore	%Cu	kt	Mt	Mt	Mt	kt	kt	kt. Oz	kt. Oz	%	%	$B
1	0.0	0.0	0.1	4.29	0.8	0.0	0.0	0.0	0.1	0.6	0.5	1.6	43.4%	32.3%	0.0
2	1.5	1.3	0.2	2.82	36.7	0.9	0.1	0.3	12.3	18.7	8.5	12.8	49.1%	32.3%	0.2
3	15.6	12.0	0.3	2.05	245.3	7.3	2.1	2.5	77.0	128.3	36.6	46.4	47.9%	32.4%	1.1
4	45.6	31.7	0.4	1.65	524.4	16.7	6.1	8.9	198.2	239.0	89.8	91.2	50.0%	32.4%	2.2
5	116.3	73.7	0.6	1.32	975.6	31.9	13.2	28.7	433.2	382.8	186.0	159.1	52.9%	32.3%	3.9
6	197.8	115.9	0.7	1.19	1,380.3	45.4	21.2	49.4	663.3	493.6	266.7	214.0	54.6%	32.3%	5.3
7	337.4	175.0	0.9	1.08	1,886.9	56.2	30.3	88.6	1,019.3	572.2	380.0	273.8	57.8%	32.2%	6.9
8	451.6	222.2	1.0	1.01	2,250.9	65.1	37.3	119.7	1,271.7	632.4	468.1	318.9	59.3%	32.2%	8.0
9	569.7	267.1	1.1	0.96	2,564.0	71.9	42.9	152.2	1,501.3	672.3	545.8	357.6	60.6%	32.2%	8.9
10	726.6	322.6	1.3	0.91	2,926.2	79.6	49.3	193.6	1,769.9	718.1	633.5	398.7	61.8%	32.2%	9.8
11	891.1	383.8	1.3	0.86	3,304.5	90.8	59.8	233.2	2,025.6	785.3	720.4	448.5	62.3%	32.2%	10.7
12	1,063.7	429.4	1.5	0.84	3,598.7	96.0	65.0	268.4	2,251.2	813.0	792.0	477.3	63.2%	32.1%	11.3
13	1,254.1	478.9	1.6	0.81	3,898.6	101.6	72.3	305.1	2,483.0	841.7	859.2	506.2	64.0%	32.1%	11.8
14	1,341.3	504.1	1.7	0.80	4,032.4	104.2	75.1	324.7	2,585.2	854.9	892.8	520.4	64.3%	32.1%	12.0
15	1,432.7	528.2	1.7	0.79	4,162.0	107.4	78.8	341.9	2,678.7	870.8	923.6	535.6	64.4%	32.1%	12.2
16	1,521.4	552.2	1.8	0.78	4,279.5	110.1	81.9	360.2	2,767.2	883.1	952.7	548.6	64.7%	32.1%	12.3
17	1,763.0	607.3	1.9	0.75	4,542.8	114.0	87.2	406.2	2,981.1	900.0	1,032.9	577.8	65.3%	32.1%	12.5
18	1,827.2	622.0	1.9	0.74	4,608.8	115.4	88.9	417.6	3,034.0	906.4	1,052.5	585.8	65.5%	32.1%	12.5
19	1,988.2	655.9	2.0	0.73	4,768.6	117.7	92.4	445.8	3,156.1	919.9	1,096.7	602.6	65.8%	32.1%	12.5
20	2,079.4	672.2	2.1	0.72	4,846.4	119.1	94.2	458.9	3,220.1	925.2	1,117.0	610.5	66.0%	32.1%	12.5
21	2,194.2	692.8	2.2	0.71	4,940.0	120.4	96.7	475.8	3,297.1	930.0	1,142.6	620.0	66.2%	32.1%	12.4
22	2,356.6	718.5	2.3	0.70	5,057.9	121.3	98.5	498.7	3,397.1	933.9	1,179.3	632.2	66.5%	32.1%	12.4
23	2,547.6	746.0	2.4	0.70	5,184.8	122.4	100.6	523.1	3,502.5	938.0	1,223.9	646.3	66.8%	32.1%	12.2
24	2,606.0	755.7	2.4	0.69	5,222.0	122.9	101.4	531.4	3,535.4	939.8	1,235.4	650.6	66.9%	32.1%	12.2
25	2,700.7	768.6	2.5	0.69	5,280.2	123.6	102.7	542.3	3,583.7	942.9	1,251.4	656.6	67.1%	32.1%	12.1
26	2,899.6	795.9	2.6	0.68	5,396.1	124.5	104.9	566.5	3,680.3	946.6	1,288.0	668.7	67.3%	32.1%	11.8
27	2,958.8	804.3	2.7	0.68	5,428.7	125.0	105.3	573.9	3,707.1	948.8	1,298.6	672.1	67.4%	32.1%	11.8
28	3,017.8	811.0	2.7	0.67	5,458.4	125.2	105.6	580.3	3,732.2	949.9	1,307.9	674.9	67.5%	32.1%	11.7
29	3,132.1	822.9	2.8	0.67	5,513.5	125.9	106.7	590.3	3,775.3	955.5	1,324.0	680.8	67.6%	32.1%	11.5
30	3,189.1	829.8	2.8	0.67	5,543.2	126.4	107.3	596.2	3,797.1	957.5	1,332.5	683.8	67.6%	32.1%	11.4
31	3,268.5	839.0	2.9	0.67	5,579.0	126.9	108.1	604.0	3,826.6	959.9	1,342.9	687.9	67.7%	32.1%	11.3
32	3,322.0	844.6	2.9	0.66	5,599.4	127.0	108.4	609.1	3,845.7	960.4	1,351.0	690.3	67.7%	32.1%	11.2
33	3,373.9	850.3	3.0	0.66	5,620.6	127.6	109.2	613.6	3,862.1	962.7	1,358.5	693.5	67.7%	32.1%	11.1
34	3,427.2	855.1	3.0	0.66	5,643.7	127.8	109.5	617.8	3,879.1	963.6	1,365.5	695.8	67.8%	32.1%	11.0
35	3,494.2	860.4	3.1	0.66	5,661.4	128.2	109.8	622.4	3,898.2	966.0	1,372.5	698.9	67.8%	32.1%	10.9
36	3,534.5	864.2	3.1	0.66	5,678.1	128.4	110.3	625.5	3,910.5	967.0	1,376.6	700.3	67.8%	32.1%	10.8
37	3,564.5	866.9	3.1	0.66	5,686.8	128.5	110.5	627.9	3,918.9	967.5	1,380.1	701.3	67.9%	32.1%	10.7
38	3,623.6	871.5	3.2	0.66	5,708.0	128.8	111.2	631.5	3,934.8	968.7	1,385.5	703.4	67.9%	32.1%	10.6
39	3,639.2	872.8	3.2	0.66	5,717.0	128.9	111.3	632.7	3,939.0	969.0	1,386.7	703.9	67.9%	32.1%	10.6

The analyses also show that the impact of increasing the royalty from 6% to 9%, ie to the now prevailing level, results in an approximate 7% reduction in undiscounted project value. Although the project value is reduced, the impact on the marginal cut-off grade is minimal, as shown in Table 15-9 (compare with the grades in Table 15-4). On this basis, there is no need to revise the optimisations, pit designs and production schedules to account for the increase. The impact could be addressed as a cashflow sensitive variable.

Table 15-7 Results of pit optimisation sensitivity analyses

Sensitivity scenario	Pit Number	Material Mined Mt	Total Ore Processed Mt	Strip Ratio Waste : Ore	Plant Feed Grade %Cu	Plant Feed Insitu Cu kt	Material Processed Oxide Plant Mt	Material Processed Mix Plant Mt	Material Processed Sulphide Plant Mt	Metal Recovered Cu in Conc kt	Metal Recovered Cathode kt	Metal Recovered Au in Conc kt. Oz	Metal Recovered Au in Dore kt. Oz	Recovery Cu & ASCu %	Recovery Au %	Pit Value $B	Change in Value %
Sell price: $2.50/lb	20	1,603.1	510.6	3.1	0.83	4,243.5	102.3	63.7	344.7	2,753.7	868.8	955.6	540.7	85.3%	64.2%	8.5	-32%
Sell price: $3.00/lb	20	2,079.4	672.2	3.1	0.72	4,846.4	119.1	94.2	458.9	3,220.1	925.2	1,117.0	610.5	85.5%	64.2%	12.5	0%
Sell price: $3.50/lb	20	2,567.7	808.7	3.2	0.66	5,296.9	133.8	108.0	566.8	3,580.5	960.1	1,250.3	662.6	85.7%	64.2%	16.9	35%
Recovery x 95%	20	2,077.0	670.6	3.1	0.72	4,841.7	108.6	103.4	458.6	3,239.9	848.5	1,125.0	602.1	84.4%	64.3%	12.2	-2%
Recovery x 100%	20	2,079.4	672.2	3.1	0.72	4,846.4	119.1	94.2	458.9	3,220.1	925.2	1,117.0	610.5	85.5%	64.2%	12.5	0%
Recovery x 105%	20	2,082.3	674.1	3.1	0.72	4,853.8	131.1	84.0	459.1	3,202.5	1,003.0	1,109.2	619.1	86.7%	64.2%	12.8	2%
Dilution = 1.00	20	2,194.4	713.2	3.1	0.73	5,235.2	123.8	99.6	489.9	3,495.2	984.9	1,212.2	658.1	85.6%	64.2%	13.8	11%
Dilution = 1.03	20	2,115.7	687.4	3.1	0.73	4,990.6	120.9	96.1	470.4	3,323.1	949.1	1,152.3	628.5	85.6%	64.2%	13.0	4%
Dilution = 1.05	20	2,079.4	672.2	3.1	0.72	4,846.4	119.1	94.2	458.9	3,220.1	925.2	1,117.0	610.5	85.5%	64.2%	12.5	0%
Dilution = 1.07	20	2,045.0	654.7	3.1	0.72	4,700.4	117.2	91.6	445.9	3,116.5	901.5	1,081.0	592.0	85.5%	64.2%	12.0	-4%
Dilution = 1.08	20	2,036.0	647.1	3.1	0.72	4,633.3	116.5	90.5	440.1	3,070.3	890.7	1,064.3	583.6	85.5%	64.2%	11.7	-6%
Dilution = 1.10	20	1,988.9	629.1	3.2	0.71	4,479.3	114.6	87.2	427.4	2,960.8	867.2	1,029.7	565.9	85.5%	64.2%	11.2	-10%
Royalty = 6%	20	2,079.4	672.2	3.1	0.72	4,846.4	119.1	94.2	458.9	3,220.1	925.2	1,117.0	610.5	85.5%	64.2%	12.5	0%
Royalty = 9%	20	2,034.3	645.8	3.1	0.74	4,772.7	116.5	90.1	439.2	3,164.8	918.5	1,097.1	601.5	85.5%	64.2%	11.6	-7%
Royalty = 20%	20	1,694.0	531.7	3.2	0.82	4,343.8	103.9	64.7	363.1	2,835.5	874.6	983.6	549.3	85.4%	64.2%	8.7	-31%
Mining costs x 95%	20	2,141.9	682.4	3.1	0.72	4,893.1	119.7	95.1	467.6	3,261.5	927.9	1,130.1	615.2	85.6%	64.2%	12.8	3%
Mining costs x 100%	20	2,079.4	672.2	3.1	0.72	4,846.4	119.1	94.2	458.9	3,220.1	925.2	1,117.0	610.5	85.5%	64.2%	12.5	0%
Mining costs x 105%	20	2,036.3	663.3	3.1	0.73	4,809.2	118.4	93.2	451.8	3,189.4	922.5	1,105.5	606.0	85.5%	64.2%	12.2	-2%
Process costs x 95%	20	2,111.7	696.9	3.0	0.70	4,906.2	122.8	97.4	476.7	3,264.3	933.8	1,132.8	618.5	85.5%	64.2%	12.8	3%
Process costs x 100%	20	2,079.4	672.2	3.1	0.72	4,846.4	119.1	94.2	458.9	3,220.1	925.2	1,117.0	610.5	85.5%	64.2%	12.5	0%
Process costs x 105%	20	2,053.6	642.4	3.2	0.74	4,779.6	116.0	88.6	437.9	3,168.6	917.8	1,099.6	601.7	85.5%	64.2%	12.2	-3%
6% recovery; cathode+con.	20	2,079.4	672.2	3.1	0.72	4,846.4	119.1	94.2	458.9	3,220.1	925.2	1,117.0	610.5	85.5%	64.2%	12.5	0%
9% recovery; cathode+anode	20	2,076.2	636.5	3.3	0.75	4,780.5	122.4	79.0	435.2	3,145.9	943.1	1,090.2	608.6	85.6%	64.2%	10.9	-13%

Table 15-8 Kansanshi marginal cut-off grades, cathode and concentrate TCRCs, 9% royalty

	Units	Float/leach Leach	Float/leach Mixed	Float Sulphide
Marginal Cut-Off Grade:				
PROCOST	$/t ore	13.54	10.82	8.88
MINDIL factor		1.05	1.05	1.05
TOTAL NET RETURN	$/10kg	51.16	47.57	54.96
C/O GRADE	**%Cu**	**0.28**	**0.24**	**0.17**

At the bottom of Table 15-7, is a comparison between the base case optimisation at 6% royalties and reflecting the sale of cathode plus concentrate, vs an optimisation at 9% royalties and reflecting the sale of cathode plus anode. These results confirm the impact of the royalty increase and the changes to the metal costs due to smelting and anode production (also accounting for 95.7% recovery to anode). Again, the impact on the marginal cut-off grades and the size of the optimal shell is minimal, and there is no need to reoptimise. The project value is reduced however, largely by virtue of the increased royalties.

15.5 Detailed pit designs

A series of phased pit designs were developed using the ultimate pit shell result (Figure 15-2 and Table 15-6) together with groupings of mining stages relating to the current mining areas (as at May 2015), and the progression of these grouped stages as determined from preliminary production schedules.

In addition to these designs, which are the basis of life of mine production scheduling (Item 16) and the Mineral Reserve estimate, a set of ultimate designs were also produced to include Inferred Mineral Resource. These designs were produced from the corresponding ultimate pit shells inclusive of Inferred Mineral Resource. In KMP operational practice, these particular designs (referred to as "M+I+I" designs) are used to comprehensively plan for the potential full extent of mining and processing and as such, allow for Inferred Resource to be anticipated in the life of mine schedule and to be drilled and reclassified in advance.

Furthermore, the M+I+I designs allow for production expansion planning, for planning of potential ultimate waste dumping extents, and for planning the location of in-pit crushers and other infrastructure.

15.5.1 **Design and planning parameters**

Drawing on the information shown in diagrams included under Item 16.3, general design parameters for haul road layouts were as follows:

- Haul road width with single conveyor = 50 m
- Haul road width to cater for trolley-assist = 50 m
- Haul road widths below in-pit crushers = 36 m
- Maximum haul ramp gradient = 1 : 10 = approximately 6°

Item 16.3 describes the lithological modelling that was undertaken to allow KMP geotechnical design criteria to be incorporated practically into the pit design process. The adopted criteria was as follows:

- Saprolite / Saprock: batter Angle = 45^{o}; berm width = 10 m; bench height = 10 m
- Clastics: batter angle = 75^{o}; berm width = 7 m; bench height = 10 m
- Marbles: batter angle = 70^{o}; berm width = 5 m; bench height = 10 m

15.5.2 **Phased and ultimate pit designs**

Main and North West Pits

Figure 15-3 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, and the location of numbered mining operational stages within the Main and North West Pits. Figure 15-4 shows Phase 1 mining and Figure 15-5 shows Phase 2 mining. These Main and North West Pit phases are not sequential in the mining sequence. Figure 15-6 shows the ultimate pit design for Mineral Reserve reporting and scheduling.

Figure 15-3 Main and North West Pits at May 2015

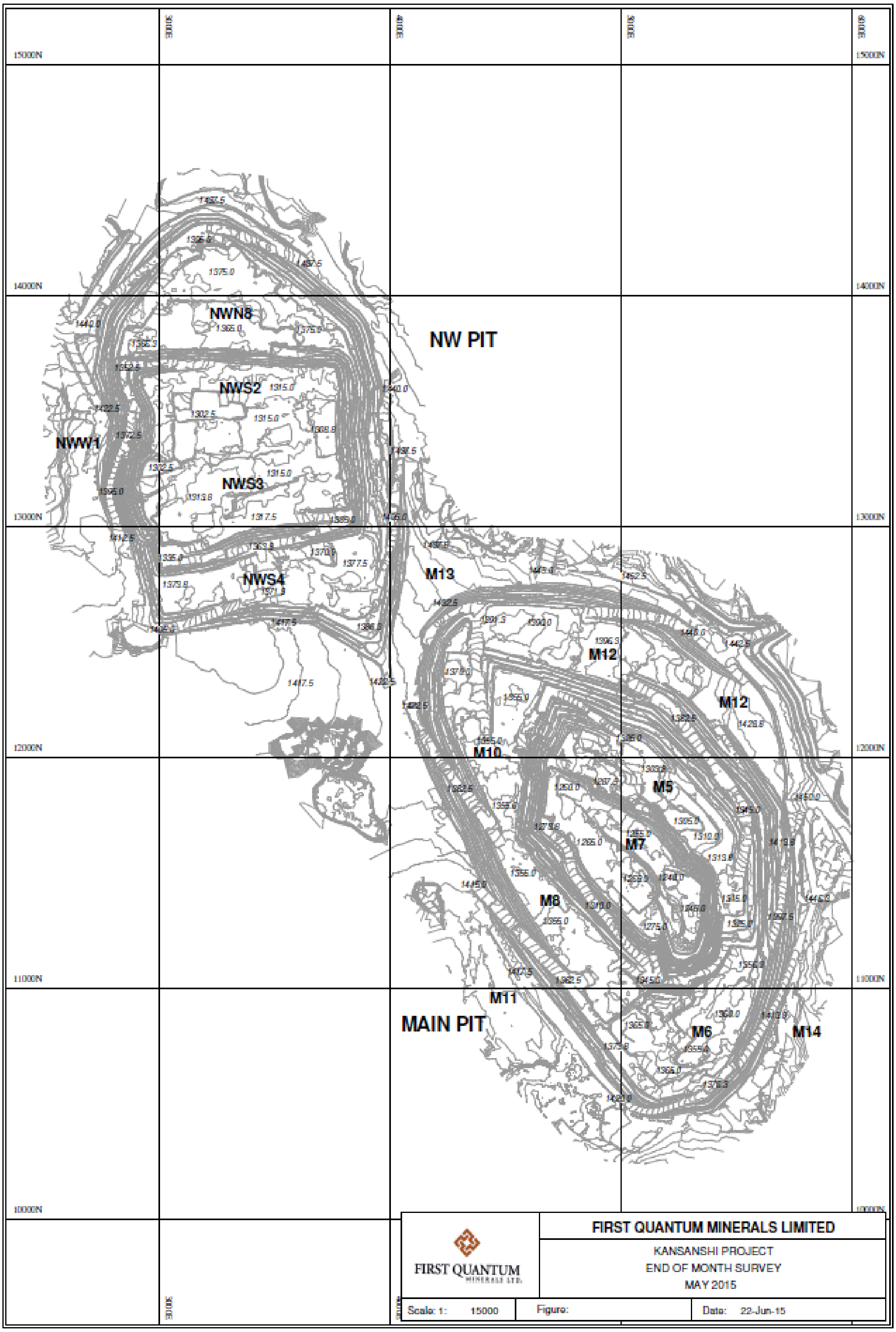


Figure 15-4 Main Pit, Phase 1

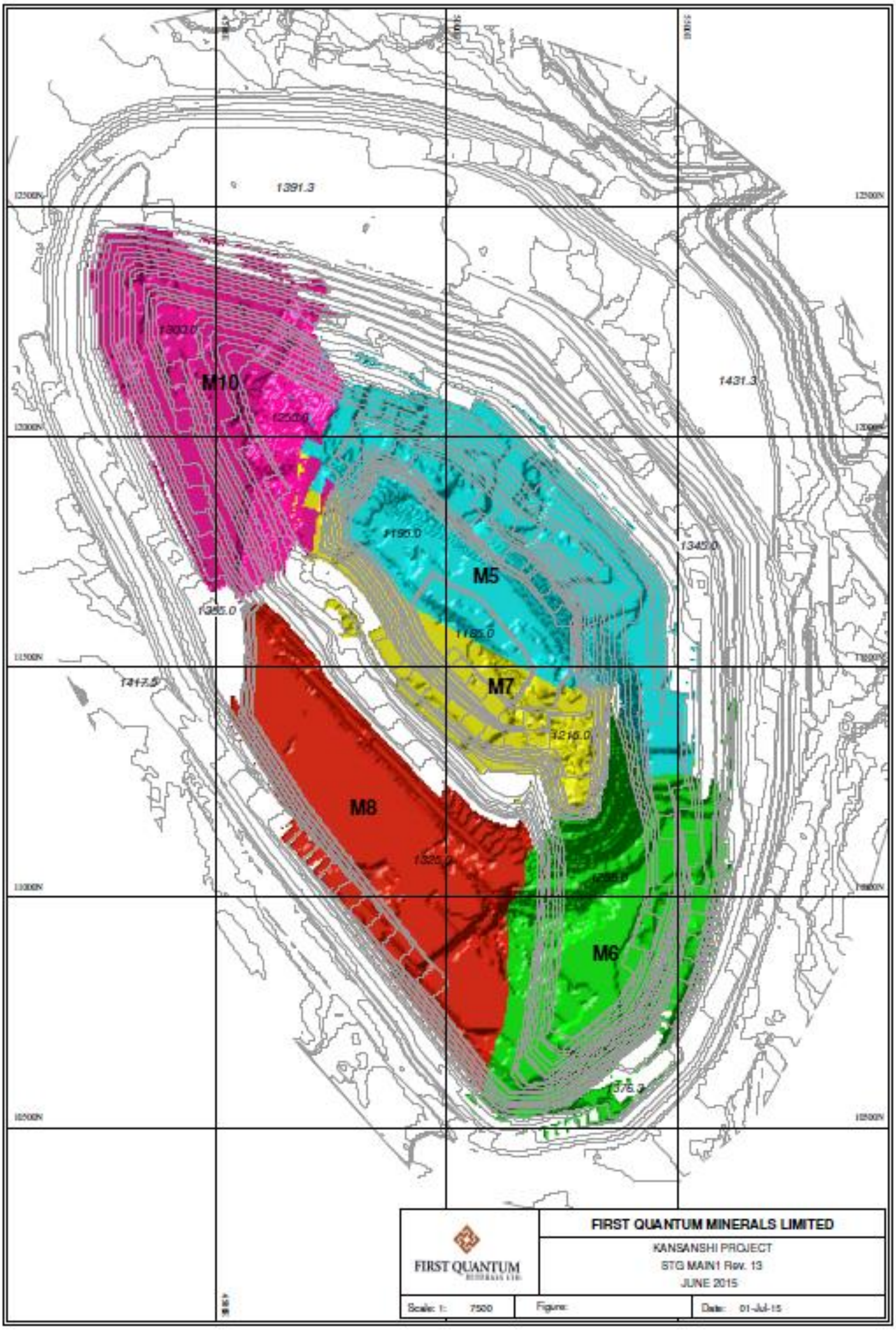


Figure 15-5 Main and North West Pits, Phase 2

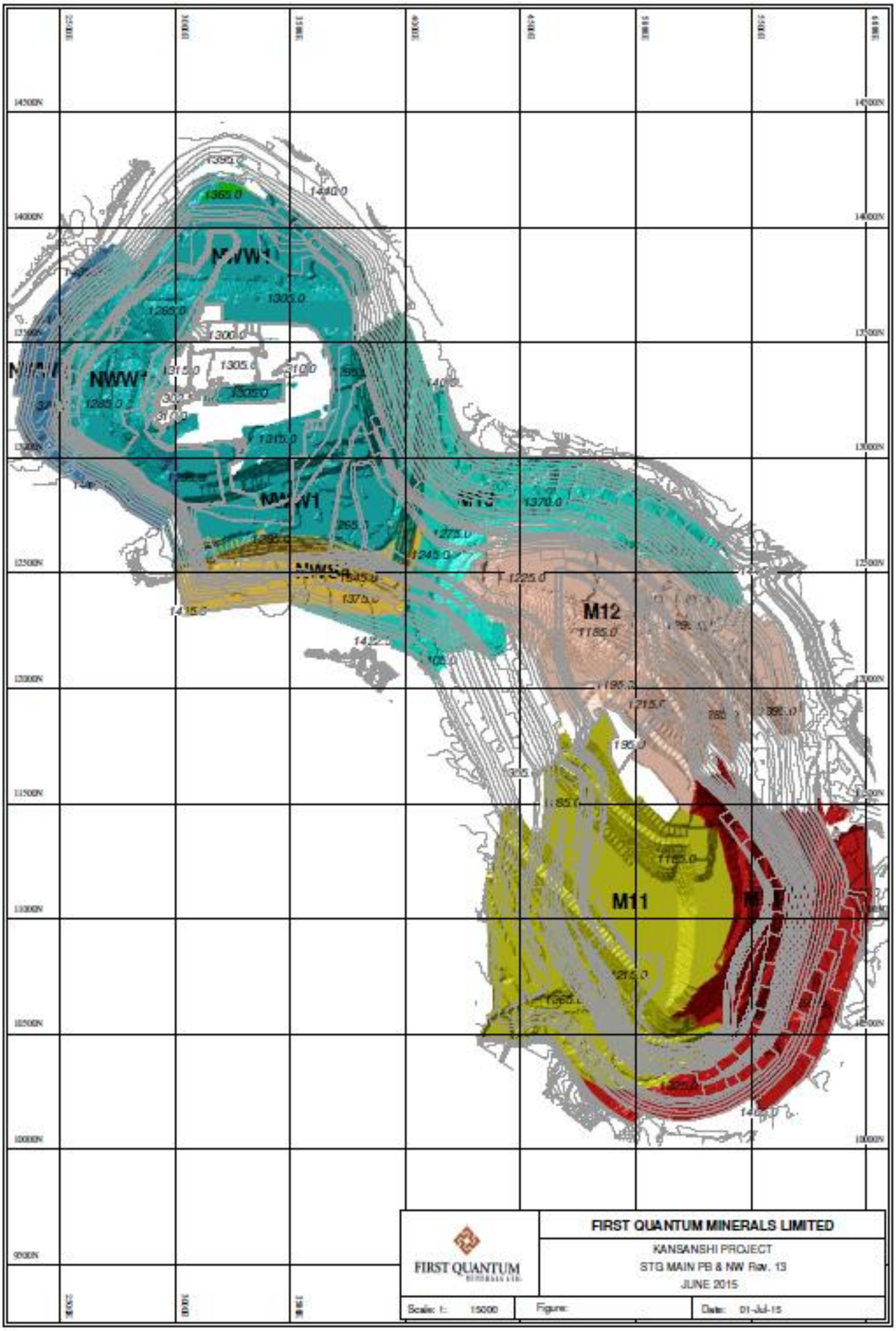


Figure 15-6 Main and North West Ultimate Pits

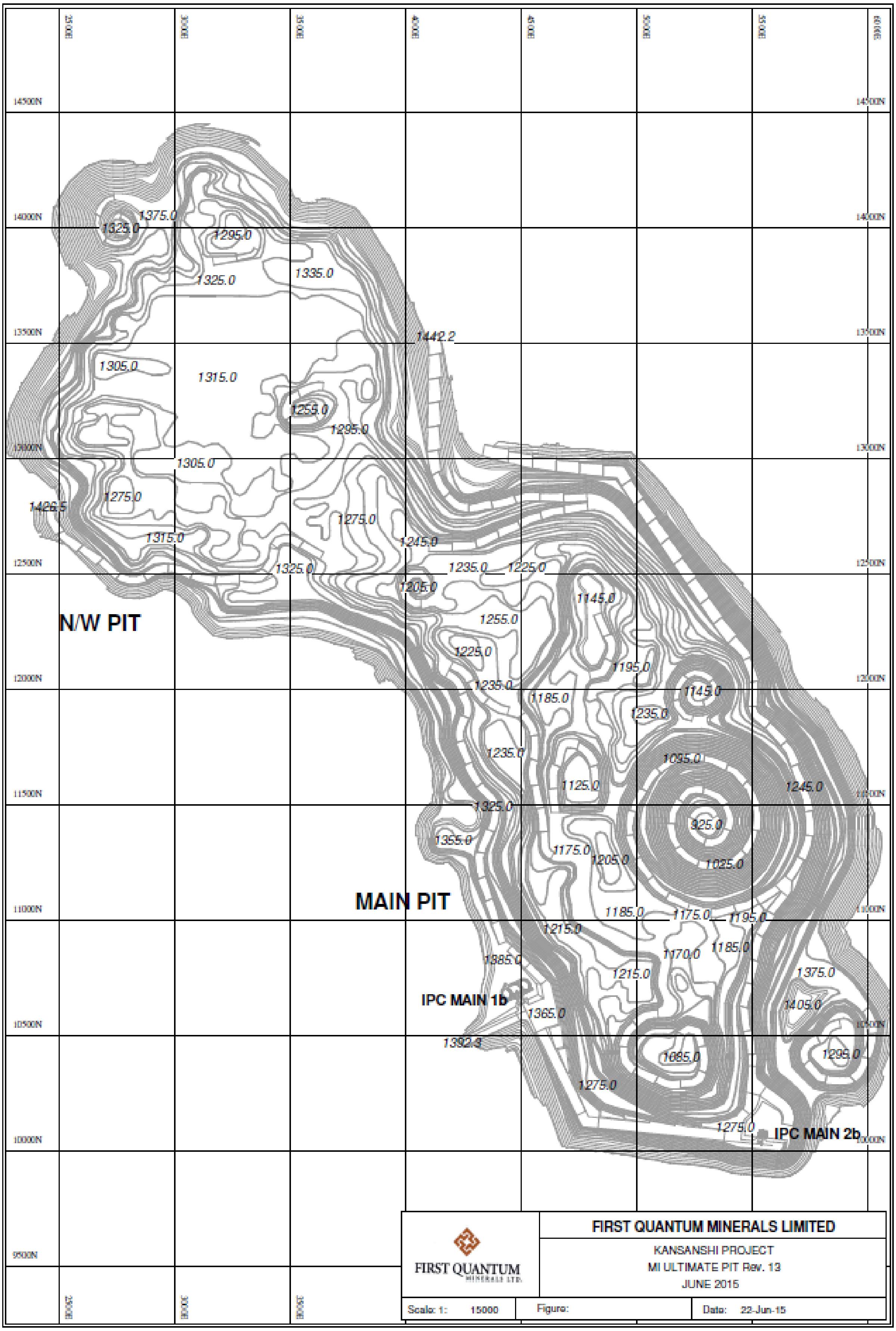


Figure 15-7 South East Dome Ultimate Pit (Mineral Reserves Pit)

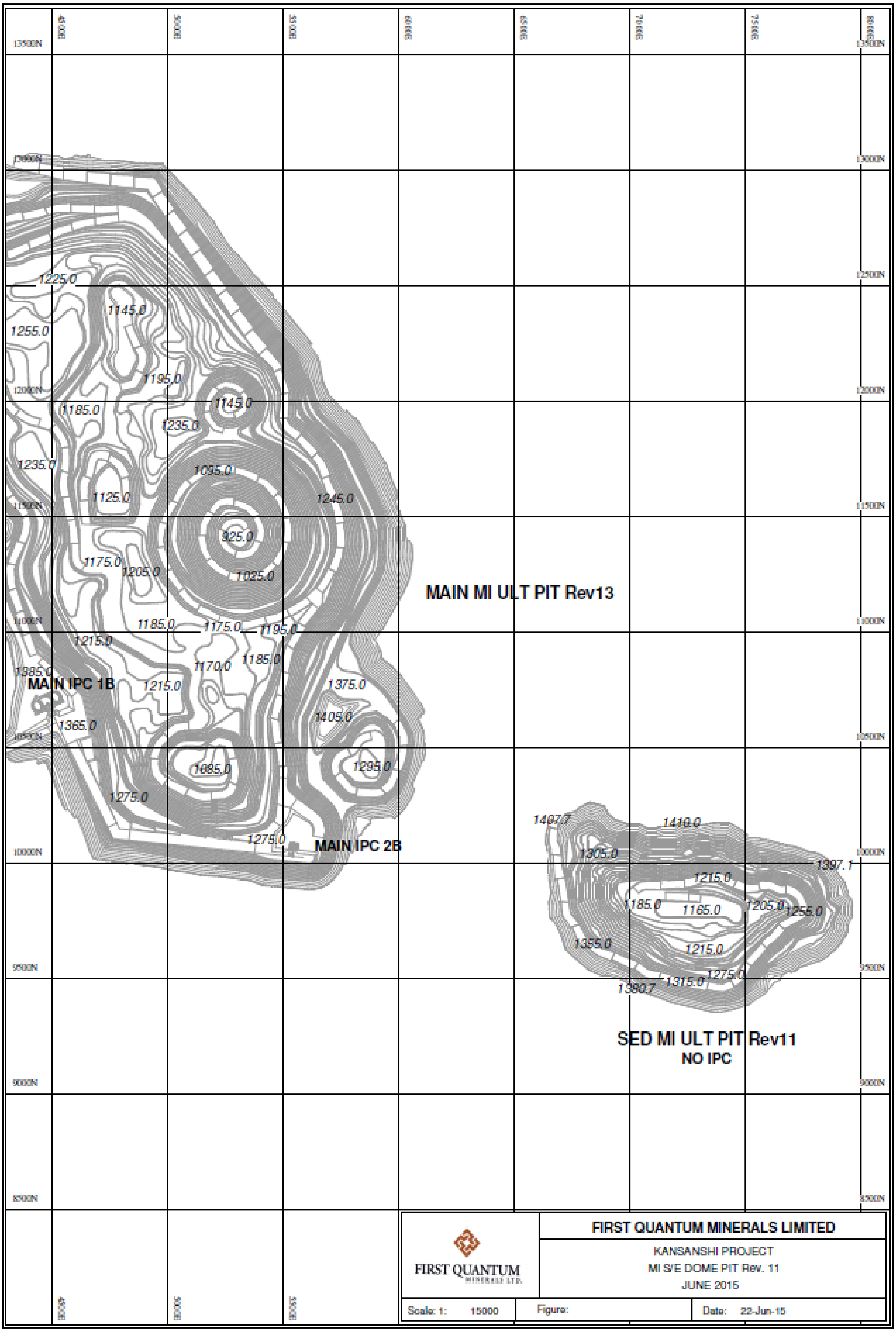
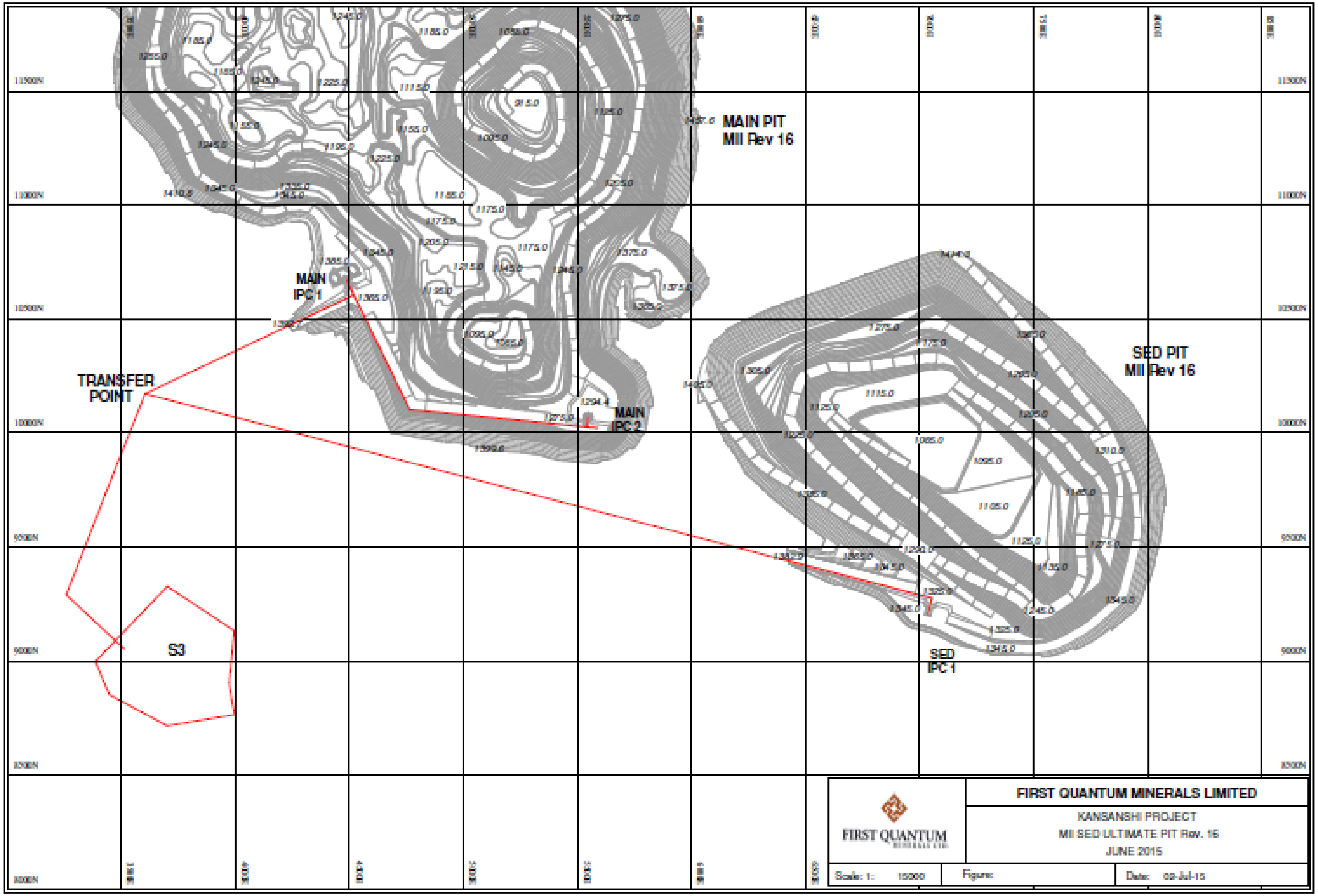


Figure 15-8 South East Dome Ultimate Pit (M+I+I Pit)



South East Dome Pit

In the case of South East Dome, there is only a proposed ultimate pit design as shown in Figure 15-7. The Mineral Reserve extents as shown are significantly less than for the M+I+I design extents, and are not favourable for the installation of an-pit crusher that would be compatible with the extension of the pit to the possible M+I+I limits. For future mine planning purposes, the M+I+I design for South East Dome includes an in-pit crusher location as shown in Figure 15-8.

15.5.3 **Ultimate pit designs, in pit crusher locations and overland conveyor route**

Figure 15-8 shows the ultimate M+I+I pit design for Main and North West, and for South East Dome. The Main Pit design shows an in-pit crusher located near surface at 1365 mRL, at a position shown as "Main IPC 1". In time, a crusher would be located deeper into the south east corner of the pit, at 1275 mRL, a position shown as "Main IPC 2". When Main IPC 2 is in place, Main IPC 1 would become an in-pit transfer point.

An in-pit crusher is also located in South East Dome, at "SED IPC 1", a position near-surface on the south wall. Discharge conveyors from Main/North West and from South East Dome are designed to converge at a common transfer point position on surface, from where crushed ore would be conveyed southwards to the S3 plant, as shown in Table 15-10.

Intermediate phases have not been designed for the M+I+I Pit, however it is conceivable that a deeper in-pit crusher could be sited to the north west of Main IPC 1 servicing the cutback to ultimate M+I+I pit limits, and conveying through an in-pit transfer point at Main IPC 1.

15.5.4 **Ultimate pit design vs optimal pit shell**

Table 15-9 shows a comparison between the selected shell 20 optimal pit inventory and the inventory for the corresponding detailed ultimate pit design. The comparison shows an acceptable overall variance.

Table 15-9 Comparison between design and optimal pit

Pit Design Ore & Waste Summary				
	Mbcm	**Mt**	**%Cu**	**Cu kt**
Ore	270.0	692.0	0.74	5,087.5
Waste	575.0	1,428.2		
Total	**845.0**	**2,120.2**		
Strip Ratio	**2.13**	**2.06**		
Optimisation (shell 20)				
	Mbcm	**Mt**	**%Cu**	**Cu kt**
Ore	262.2	672.2	0.72	4,846.4
Waste	566.6	1,407.2		
Total	828.8	2,079.4		
Strip Ratio	**2.16**	**2.09**		
Comparison (variance): Design vs Optimisation				
	Mbcm	**Mt**	**%Cu**	**Cu kt**
Ore	103%	103%	102%	105%
Waste	101%	101%		
Total	**102%**	**102%**		

15.5.5 Ultimate pit design depleted for production to date

Before proceeding with the Mineral Reserves estimate and production scheduling (Item 16), the detailed ultimate pit design inventory was depleted to reflect updated mining activity to the end of May 2015. The depleted inventory is listed in Table 15-10.

Table 15-10 Design pit inventory accounting for current depletion

	Mbcm	Mt	%Cu	Cu kt
Ore				
Main & NW Pits	239.4	613.5	0.73	4,467.2
South East Dome	21.9	56.2	0.83	464.5
subtotal	**261.3**	**669.7**	**0.74**	**4,931.6**
Waste				
Main & NW Pits	508.5	1,263.0		
South East Dome	60.6	150.6		
subtotal	**569.1**	**1,413.5**		
Total				
Main & NW Pits	747.9	1876.5		
South East Dome	82.5	206.7		
subtotal	**830.4**	**2,083.3**		
Strip Ratio				
Main & NW Pits	2.12	2.06		
South East Dome	2.77	2.68		
subtotal	**2.18**	**2.11**		

15.6 Mineral Reserve statement

The Mineral Reserve within the designed pits is estimated as 668.0 million tonnes at an average grade of 0.70% TCu. The estimate is entirely within the Measured and Indicated Mineral Resource reporting envelope listed in Table 14-14. A breakdown by pit and classification is provided in Table 15-11.

The reported Mineral Reserve is based on an economic cut-off grade which accounts for longer-term copper metal and gold price projections of \$3.00/lb (\$6,615/t) and \$1,200/oz, respectively. The inventory reflects the phased pit designs and the mining production schedule described in Item 16.5.

The Mineral Reserve listed in Table 15-11 represents mining recovery of 80% and 77% of the Measured plus Indicated copper Mineral Resource tonnage and insitu metal estimate, respectively. In total, 16.6% of the combined Mineral Reserve tonnage is classified as Proven with the remainder classified as Probable.

Table 15-12 lists the additional Mineral Reserve held in surface stockpiles.

Table 15-11 Kansanshi Operations Mineral Reserves statement, in-pit inventory at May 2015

MINERAL RESERVE AT 31st MAY 2015 (AT $3.00/lb Cu and $1200/ounce Au)							
Leach Ore (Float/leach feed)							
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**	**TCu metal (kt)**	**Au metal (t)**
MAIN & NW	Proved	24.1	1.13	0.71	0.13	273	3.2
	Probable	88.1	1.22	0.76	0.14	1,073	12.5
	Total P+P	**112.1**	**1.20**	**0.75**	**0.14**	**1,346**	**15.7**
SE DOME	Proved						
	Probable	3.8	0.89	0.40	0.14	33	0.5
	Total P+P	**3.8**	**0.89**	**0.40**	**0.14**	**33**	**0.5**
TOTAL	Proved	24.1	1.13	0.71	0.13	273	3.2
	Probable	91.8	1.20	0.75	0.14	1,106	13.0
	Total P+P	**115.9**	**1.19**	**0.74**	**0.14**	**1,379**	**16.2**
Mixed Ore (Float/leach feed)							
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**	**TCu metal (kt)**	**Au metal (t)**
MAIN & NW	Proved	15.3	0.54	0.13	0.09	82	1.4
	Probable	68.1	0.73	0.16	0.13	495	8.5
	Total P+P	**83.4**	**0.69**	**0.15**	**0.12**	**577**	**10.0**
SE DOME	Proved						
	Probable	5.5	0.65	0.15	0.12	36	0.7
	Total P+P	**5.5**	**0.65**	**0.15**	**0.12**	**36**	**0.7**
TOTAL	Proved	15.3	0.54	0.13	0.09	82	1.4
	Probable	73.6	0.72	0.16	0.12	530	9.2
	Total P+P	**88.9**	**0.69**	**0.15**	**0.12**	**613**	**10.6**
Sulphide Ore (Float feed)							
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**	**TCu metal (kt)**	**Au metal (t)**
MAIN & NW	Proved	71.2	0.49	0.02	0.11	352	7.6
	Probable	345.3	0.57	0.02	0.12	1,969	40.6
	Total P+P	**416.5**	**0.56**	**0.02**	**0.12**	**2,321**	**48.3**
SE DOME	Proved						
	Probable	46.8	0.80	0.03	0.13	372	6.2
	Total P+P	**46.8**	**0.80**	**0.03**	**0.13**	**372**	**6.2**
TOTAL	Proved	71.2	0.49	0.02	0.11	352	7.6
	Probable	392.1	0.60	0.02	0.12	2,341	46.8
	Total P+P	**463.2**	**0.58**	**0.02**	**0.12**	**2,693**	**54.4**
Total Ore							
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**	**TCu metal (kt)**	**Au metal (t)**
MAIN & NW	Proved	110.6	0.64	0.18	0.11	707	12.3
	Probable	501.4	0.71	0.17	0.12	3,537	61.7
	Total P+P	**612.0**	**0.69**	**0.17**	**0.12**	**4,244**	**73.9**
SE DOME	Proved						
	Probable	56.0	0.79	0.06	0.13	441	7.3
	Total P+P	**56.0**	**0.79**	**0.06**	**0.13**	**441**	**7.3**
TOTAL	Proved	110.6	0.64	0.18	0.11	707	12.3
	Probable	**557.4**	**0.71**	**0.16**	**0.12**	3,978	69.0
	Total P+P	**668.0**	**0.70**	**0.16**	**0.12**	**4,685**	**81.3**

Table 15-12 Kansanshi Operations Mineral Reserves statement, stockpile inventory at May 2015

MINERAL RESERVE AT 31st MAY 2015 (AT $3.00/lb Cu and $1200/ounce Au)							
Leach Ore (Float/leach feed)							
S/Piles	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**	**TCu metal (kt)**	**Au metal (t)**
TOTAL	Proved						
	Probable	0.0	0.00	0.00		0	
	Total P+P	**0.0**	**0.00**	**0.00**		**0**	
Mixed Ore (Float/leach feed)							
S/Piles	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**	**TCu metal (kt)**	**Au metal (t)**
TOTAL	Proved						
	Probable	46.4	0.55	0.23		256	0.0
	Total P+P	**46.4**	**0.55**	**0.23**		**256**	
Sulphide Ore (Float feed)							
S/Piles	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**	**TCu metal (kt)**	**Au metal (t)**
TOTAL	Proved						
	Probable	7.2	0.36	0.02		26	0.0
	Total P+P	**7.2**	**0.36**	**0.02**		**26**	
Total Ore							
S/Piles	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**	**TCu metal (kt)**	**Au metal (t)**
TOTAL	Proved						
	Probable	53.6	0.53	0.20		282	0.0
	Total P+P	**53.6**	**0.53**	0.20		282	0.0

ITEM 16 MINING METHODS

The following information is reproduced in part from the December 2012 Technical Report (Journet and Cameron, 2012), with updates provided by Michael Lawlor (QP) and including contributions from KMP staff.

16.1 Mining details

16.1.1 Mine site layout

The current layout of the Kansanshi Operations site is shown in Figure 16-1. The main features shown are:

- the existing North West and Main Pits, and the location of the proposed South East Dome Pit (M+I+I design shown)
- the location of the S3 expansion project, relative to the existing process plant
- the location of the smelter
- the location of the overland ore conveyor transfer station to the S3 plant
- the location of the mine workshops

16.1.2 Mining method and operations

Open pit mining at Kansanshi is based on conventional drill and blast, shovel and truck mining techniques. Mining has proceeded from initial excavations in two pits (Main and North West) through a sequence of cutbacks, which in the longer term will result in these pits merging. A newly defined, nearby resource at South East Dome will contribute to the longer term production profile, but is unlikely to merge with the other two pits.

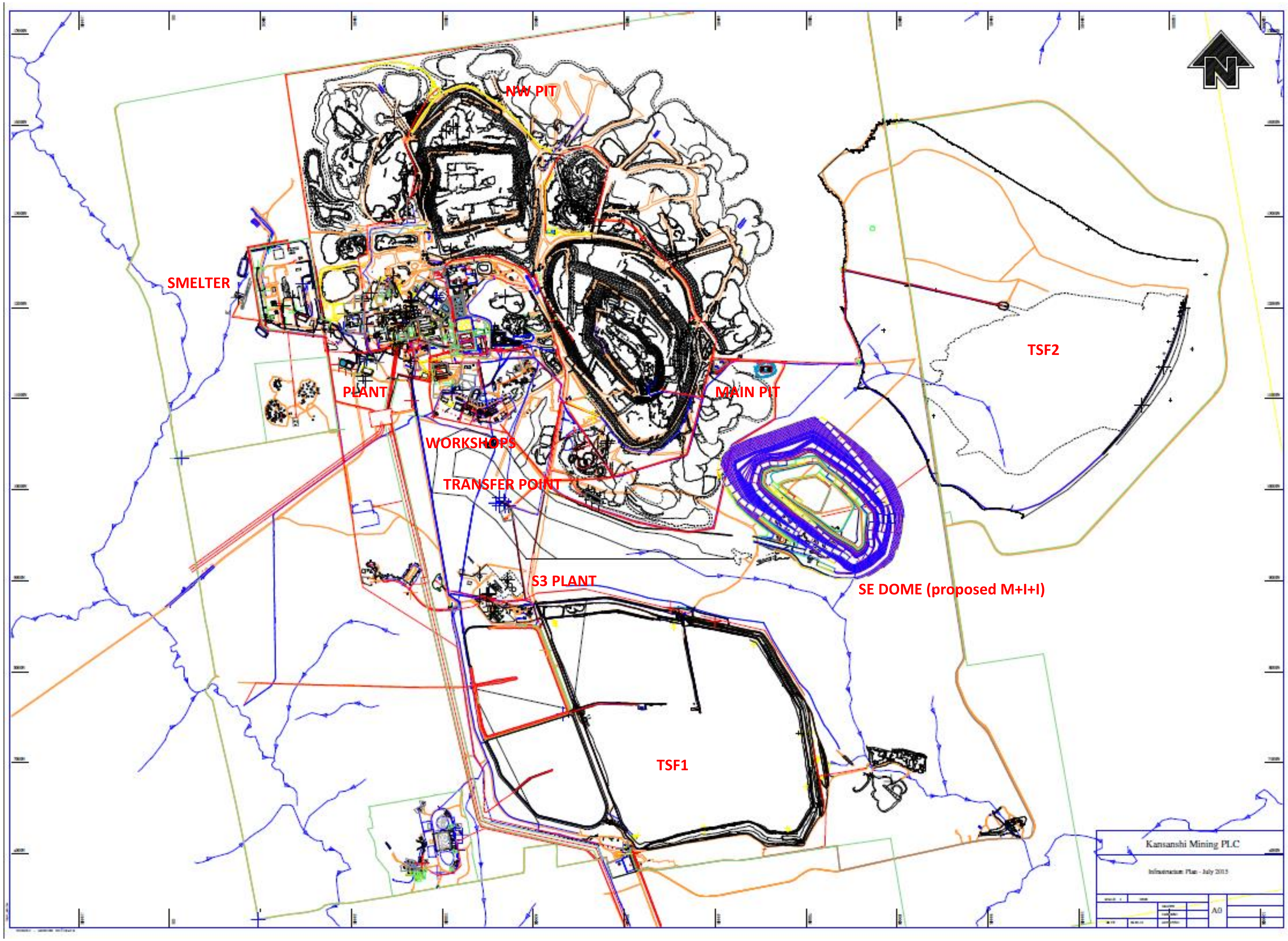
The cutbacks generally comprise wide benches of 200 m to 300 m width, providing several mining horizons to satisfy the feed requirements for multiple processing routes. In general, ore is hauled to a ROM Pad located immediately south of the North West Pit, whereas waste is hauled to various dumps:

- around the northern and southern extremities of the Main Pit
- to around the western, northern and eastern extremities of the North West Pit
- to the northern, eastern and southern extremities of the future South East Dome Pit

The prime focus for future mining is to maximise the use of grid electrical power throughout the mining process, coupled with bulk mining systems to ensure that unit costs can be minimised. Initially, this strategy has been met with the installation of trolley-assist equipment for a portion of the truck fleet.

To accommodate the increased processing requirements for the S3 expansion, the existing mining fleet will be expanded. New cutbacks for the Main and North West Pits have been designed to accommodate not only trolley-assist waste haulage routes but also proposed semi-mobile, in-pit crusher locations and associated conveyor routes.

Figure 16-1 KMP site layout plan



16.1.3 Mining fleet

The Kansanshi mining fleet comprises diesel powered production drills, various sized hydraulic shovels/excavators and a variety of haul truck makes and capacities. The ore mining is carried out by the Company's own equipment fleet operating under the entity of First Quantum Minerals Operations (FQMO).

The current primary and auxiliary equipment fleet inventory is listed in Table 16-1.

Table 16-1 Kansanshi Operations, mining equipment fleet

Drills	**#**	**Track dozers**	**#**
Pantera DP1500i	5	Cat D10T	2
Cubex QXR1120 Gen V	3	Komatsu D275A-5R	2
Drill Tech D25KS	4	Komatsu D375A-5/6	9
Drill Tech D45KS	2	Komatsu D475A-5	4
Cubex DR560	8	**Wheel dozers**	**#**
Excavators/shovels	**#**	Cat 834G	2
Hitachi ZX870LCH	1	Komatsu WD900-3	4
Volvo EC290BC	1	**Front-end loaders**	**#**
Volvo EC360BLC	1	Volvo L120F	2
Volvo EC300DL	3	Hitachi LX170E	1
Volvo EC700CL	2	Hitachi LX290E	1
Hitachi EX1900BH	2	Cat 992G Loader	1
Hitachi EX2500BE	3	Komatsu WA500-6	2
Liebherr R9250D BH	2	Komatsu WA900-3	1
Liebherr R9350ER	4	**Graders**	**#**
Liebherr R9350D BH	3	Komatsu GD675-5	2
Liebherr R984C	5	Komatsu GD825A-2	10
Trucks	**#**	Caterpillar 16H	2
Volvo A40 D/F	11	**Fuel, service and water trucks**	**#**
Cat 777D	15	Volvo A40D - Fuel Truck	6
Cat 785	13	Volvo A40D - Service Truck	10
Euclid EH3500ACII	38	Volvo A40D/F - Water Truck	10
Komatsu HD785-5	7	Komatsu HD785-7 WB	2
Komatsu HD1500-7	3		

In addition to the equipment listed in Table 16-1, ancillary mining equipment includes:

- integrated tool carriers
- stemming loaders
- cable reeler
- tyre handlers
- telehandlers
- forklifts
- crane
- backhoe
- compactors

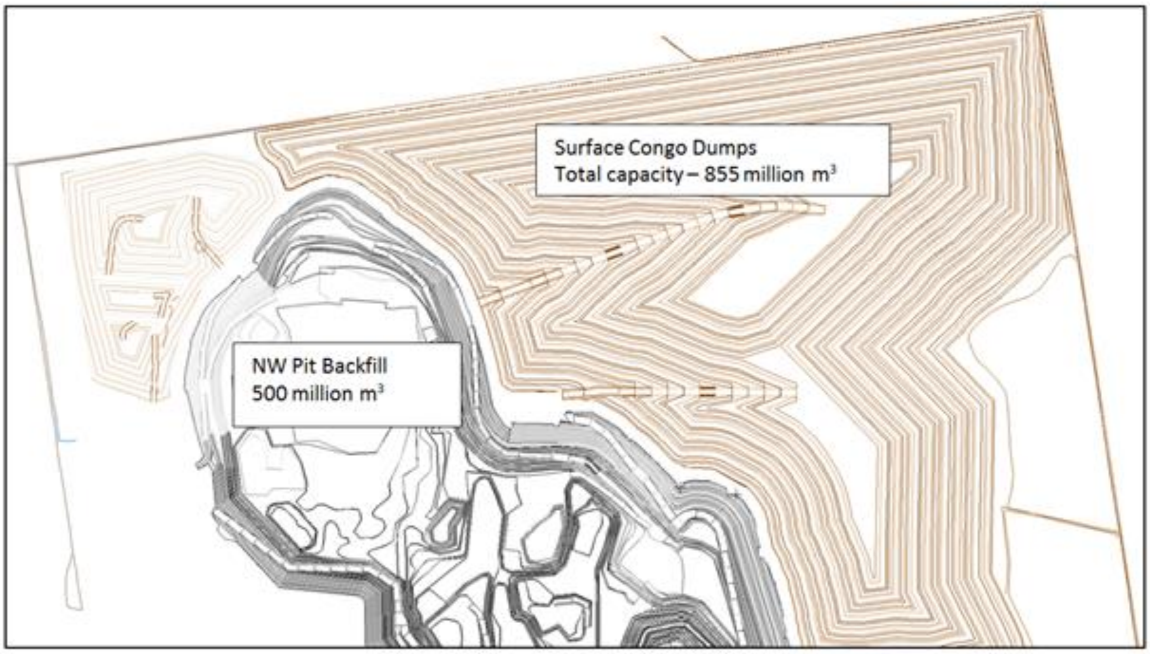
Towards the end of 2009 an electrification programme was implemented at Kansanshi to supplement the existing diesel-electric trucks with an AC-drive fleet fitted with trolley-assist (TA) pantographs. At this time, a near-surface length of inclined haulroad on the west side of the Main Pit had been equipped with cantilevered catenary wires for TA trucks destined for the waste dump. As of May 2015, two additional cantilevered catenary wires for TA trucks have been installed at KMP, one on the ramp leading to the Congo waste dump (Item 16.1.4) on the north western edge of the Main Pit, and the other on the Main 10 access ramp. It is expected that the TA haulage routes will be significantly increased for the Main and North West Pit pushbacks supporting the S3 expansion project production requirements.

16.1.4 **Waste dumping**

Active waste dumps are located predominantly around the eastern flank of the Main and North West Pits. The landform concept was originally developed as part of a study carried out by DumpSolver in 2006. Current waste dumping activity lies within the dump footprints identified from this particular study.

Within the unexpanded LML area, the dumps on the eastern flank of the pits can be extended to the eastern boundary of the licence (Figure 16-2, referred to as the “Congo” dumps). There is limited capacity on the west side of the North West Pit and the existing dumps to the south and south east of Main Pit would need to be moved to accommodate mining to the M+I+I design limits (Figure 16-1).

Figure 16-2 Waste dump



Capped at 1500 mRL, the Congo dumps have a volume capacity of 855 million m^3. This can be expanded if necessary through an increase in height or in footprint, by extending eastwards into the enlarged LML license area. Following completion of mining in the North West Pit, the pit backfill volume capacity is a further 500 million m^3.

Upon completion of the North West Pit, mined waste material could be hauled to either the Congo dumps or to backfill, depending upon minimum haulage distances. Conceptually, all near-to-surface material from the Main Pit pushbacks could be hauled to the Congo dumps while in-pit waste from lower levels could be hauled to backfill. All waste from South East Dome could be hauled to the Congo dumps. A waste dumping and backfill schedule based on the LOM production schedule is provided in Item 16.5.2.

Despite the presence of sulphide minerals that exist in fresh waste rock mined at Kansanshi, there is no evidence to suggest that the waste rock is acid generating when exposed to oxygen and water. This is due to the existence of acid-neutralising carbonates within the waste material.

16.1.5 **Trolley-assisted haulage**

The trolley-assist system has been installed on inclined ramps within the Main Pit above 1355 mRL, and leading to the surface waste dumps. The trolley-assist system will be extended onto inclined haul roads on the waste rock dumps. The transition between non-trolley and trolley-assist ramps will be achieved with 100 m long horizontal haul road sections to minimise wheel torque damage to the road surface due to rapid acceleration and power transfer associated with electric power.

Apart from allowing for sufficiently wide and constant gradient haul ramps in the Life of Mine design, the detailed design and costing for the trolley-assist layout has not yet been carried out.

16.1.6 **In-pit crushing and conveying**

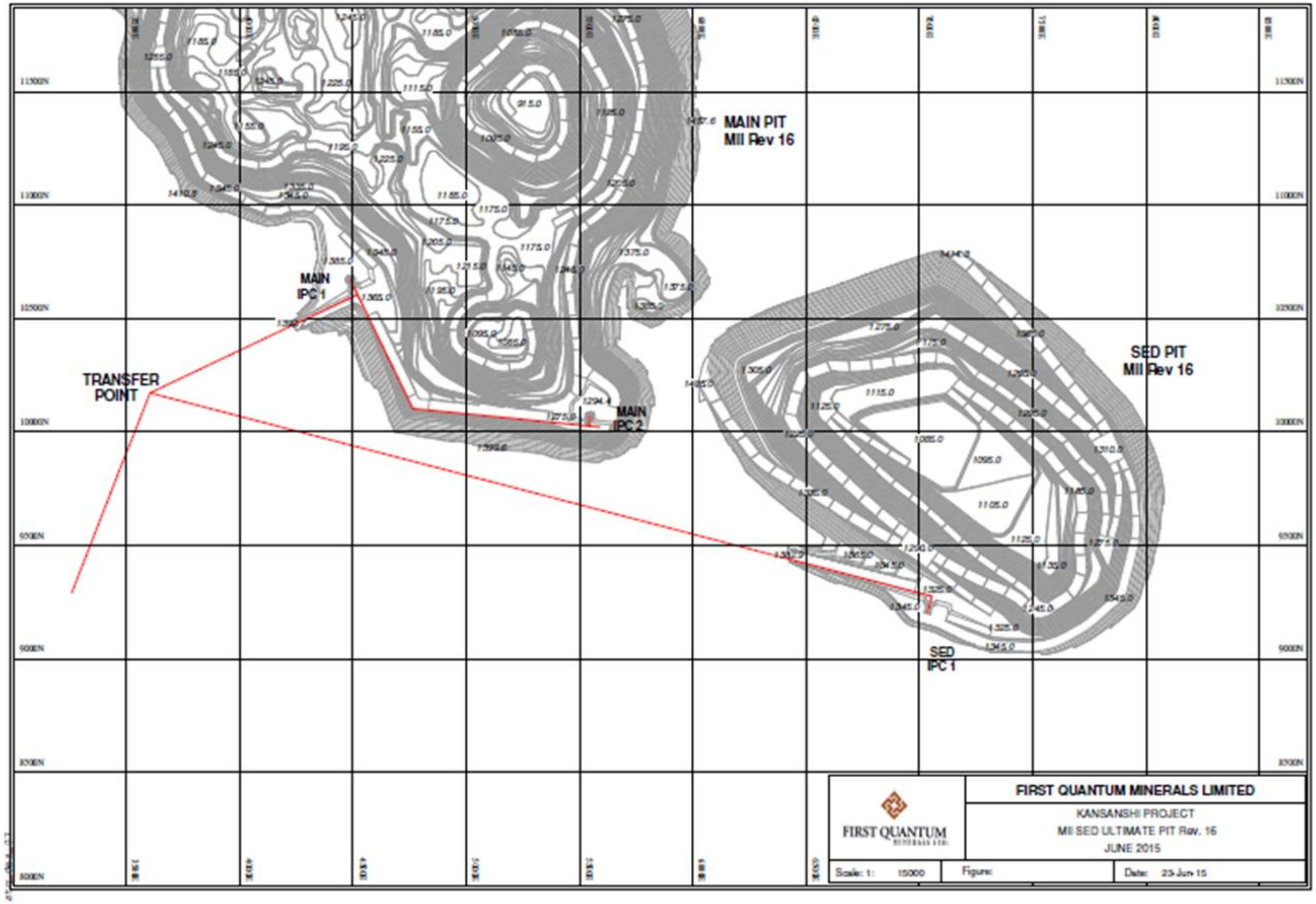
It is proposed that two in-pit crushers of similar specification (ThyssenKrupp KB63-89) to those specified for the Company's Sentinel operation, will be installed at Kansanshi within the current five-year mine plan timeframe, and at suitable locations in the Main Pit and/or the South East Dome Pit. Crushed ore will be conveyed to a surface transfer bin and then to the plant by overland conveyors. The location of the initial crusher pockets are shown in Figure 16-3.

Based on the Project schedule requirements as summarised in Section 11.0, the excavations and preparation of both crusher sites would need to be in place by the end of 2016 in Main Pit and by the end of 2019 in the South East Dome Pit.

Design of the in-pit crushing and conveying (IPCC) system has taken account of the following:

- Desired locations for the surface overland conveyor routes and transfer points.
- Installations are to be located on the southern side of the Main and North West Pits, thereby enabling unimpeded trolley-assist waste haulage routes on the northern side of these pits.
- The initial locations should be, ideally, in fresh rock horizons to allow the excavation of stable crusher pockets.
- Where possible, subsequent crusher positions must provide for ease of crusher relocation and logical extension of conveyor belt infrastructure.
- As pushbacks develop towards the ultimate limits, the in-pit crushers will need to be relocated to near-surface positions, to avoid situations where production blasting is above the installations.
- In-pit transfer points and haul road/belt crossovers are to be minimised.
- The largest truck which could present to the crushers is a Hitachi EH3500 truck (8.1 m width).

Figure 16-3 Proposed initial in-pit crusher pocket locations



16.1.7 **Ore stockpiles**

The production schedule (and Mineral Reserves statement, Table 15-13) include the reclaim and processing of ore on current stockpiles located around the rim of the North West Pit and adjacent to the run-of-mine (ROM) pad and primary crushers.

The production schedule also features the building and reclaim of float/leach feed stockpiles reaching a size of 81.6 Mt and a sulphide feed stockpile reaching a size of 76.4 Mt (Item 16.5.1). The long-term float/leach stockpiles will need to be located on top of the waste dumps, from where ore can be reclaimed and hauled across to the ROM pad adjacent to the existing plant. Some of the sulphide feed (from Main and South East Dome) could be hauled to stockpiles located adjacent to the surface transfer point from where the ore could be reclaimed and hauled a short distance back to Main IPC 1 crusher (near-surface). Other sulphide ore will need to be stockpiled on top of the waste dumps and also reclaimed and hauled across to the surface ROM pad and associated “gyro” crusher.

16.2 **Mining sequence**

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

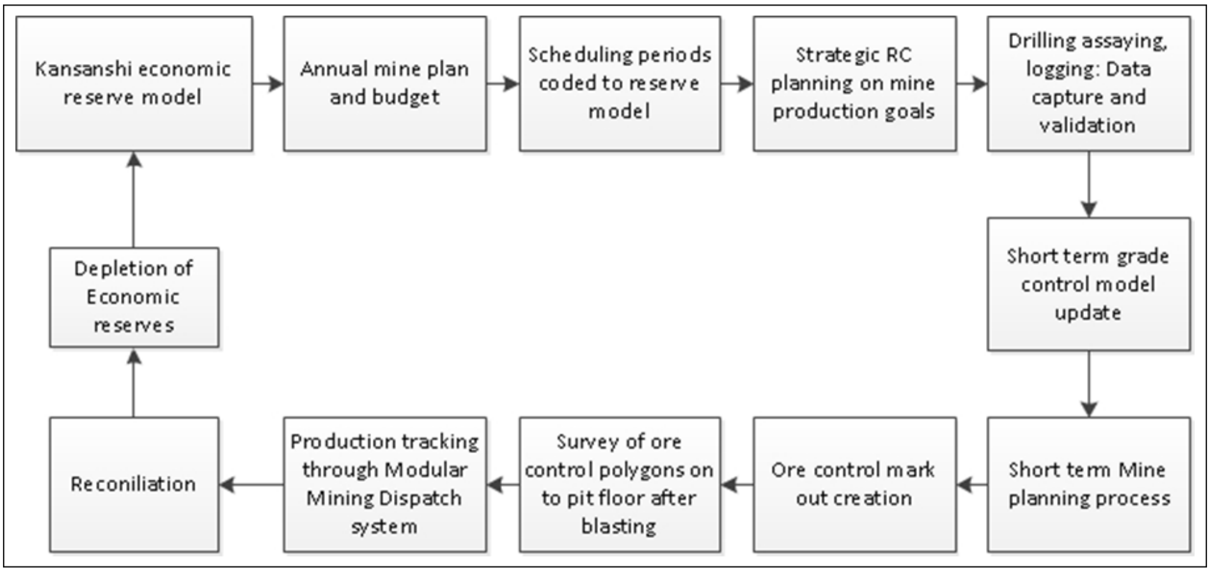
- RC grade control drill holes delineate the ore zones.
- Blast patterns are designed to reduce material throw and ore dilution – and a Blast Master planning process controls the sequence of operation.
- Ore and waste are blasted and mined separately as fragmentation requirements can vary significantly. Ore fragmentation generally requires a higher powder factor.
- Waste is removed on each 10 m bench subsequent to the mining of ore.
- The removal of waste in the successive cut-backs utilises bulk systems of operation.
- Trim blasts and perimeter blasting 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 rock into a mixed fleet of 180 tonne to 95 tonne capacity haul trucks.
- Ore is hauled from the pit to respective crushers and is preferentially tipped directly into the appropriate crusher. Finger stockpiles are however an integral part of the feed sequence to ensure ore blending can be achieved, haulage efficiencies can be maximised and operational flexibility enhanced at all times. Presently interim stockpiles are used. This practice is to be minimised as it can introduce triple handling of ore feed and thus increased costs.
- De-bottlenecking and improved management of the ROM pad has been implemented to improve run-of-mine ore handling efficiency.
- From 2017, it is expected that sulphide ore will be short-hauled to the in-pit crushers.

16.3 **Grade control process**

The Kansanshi mine grade control process is driven by the annual budgeting and short term mine planning process. Short term mine planning is defined as a time period more or less equal to twelve months. A schematic process flow is presented in Figure 16-4 to assist in the explanation that is detailed below. Figure 16-5 provides a more detailed explanation with specific reference to geological data management processes.

Based on the Kansanshi Mineral Reserve model, a mine schedule is created by the planning team as part of the annual budgeting process. This process is usually completed in in November and the schedule created runs for 14 months up to the end of the following year. From the schedule a set of one month periods are written to the block model using Surpac to provide a spatial and time based constraint.

Figure 16-4 Schematic of the Kansanshi mine grade control process with reference to short term planning

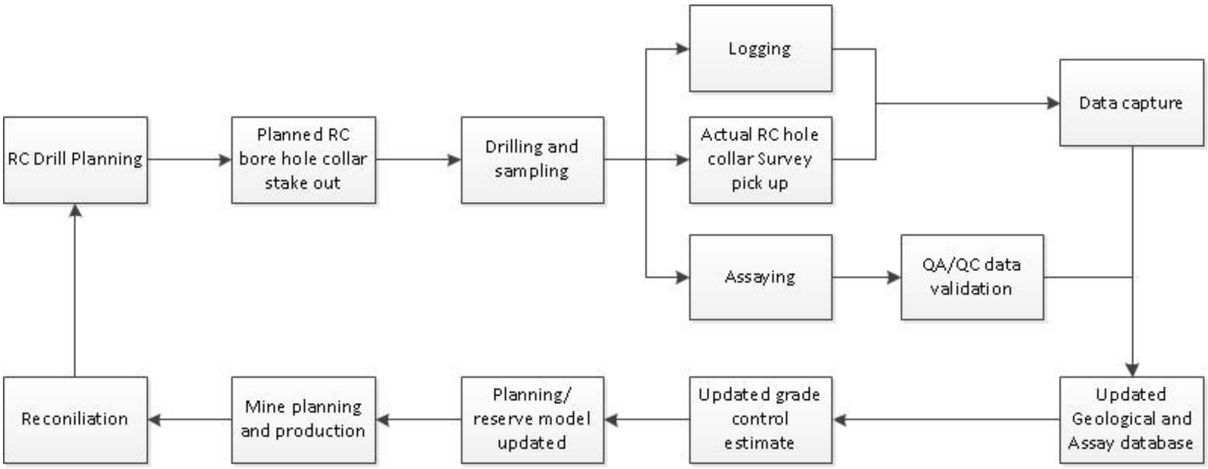


The mine utilises reverse circulation (RC) drilling to collect 3 m composite geological samples on 12.5 m (east/west) by 15 m (north/south) grid spacing and at a 60° inclination. The 3 m composite samples are split down to an appropriate size before submission for assay. Assays are performed on these samples to quantify the following:

- Total Copper Percentage – TCu (%)
- Acid Soluble Copper Percentage – AsCu (%)
- Gold content – Au (ppm) (every fourth hole)
- Nickel Percentage – Ni (%) (every fourth hole)
- Gangue Acid Consumption – GAC (kg/t)

Using the planning periods from the mine schedule, the model is constrained and sampling coverage for those areas is assessed. Strategic and operational plans are developed to ensure that RC drill coverage of these areas is completed up to no less than three months ahead of the mining face.

Figure 16-5 Overview of the geological data acquisition, management and ore control process flow utilised at Kansanshi mine



Assaying is completed by an onsite laboratory and by an external laboratory service. Approximately 25% of samples are sent to the external lab. Both geological logging and assay data is captured/ imported into, validated and stored in a DataShed™ geological and assay database system. Analytical methods employed are the following for:

- TCu – Multi acid digest with AAS finish.
- AsCu – Single acid digest with AAS finish.
- Au – lead collection fire assay, followed by multi acid digest with AAS.

Geological logging data is utilised to update geological wireframe surfaces that demarcate stratigraphy, weathering and oxidation. Assay data is then estimated into these domains/ volumes. Data updates of assay data occur on a monthly basis, however updates of wireframes for estimation occur on a quarterly basis as they require a greater amount of time to complete the validation process.

The updated grade control model is provided to the mine planning team complete with new RC assay data. An ore classification macro is applied to categorise the main ore types and assist with the short term mine planning process. Areas of ore demarcated for production drilling and blasting are defined using a polygon boundary. These boundaries are then used by the mine geology team to create a geological ore control mark-out that has economic factors applied to it throughout the mine plan. These polygons are surveyed on to the pit floor to guide the mining process and feed in to production tracking and reconciliation. These ore polygons are also published to the dispatch system and are used for ore tracking purposes.

16.3.1 Drilling and blasting

Saprolite material is generally mined as free-dig. Elsewhere, a fleet of Ingersoll Rand DK45 and DK25 drills, and Cubex 560 and 1120 drills, supplemented with Pantera 1500 rigs are currently used for production drilling. Powder factors are in the range of 0.88 to 1.10 kg/m^3 (0.28 to 0.32 kg/t).

Perimeter blasting is undertaken on final walls to prevent blast damage and to maintain wall control.

16.4 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.

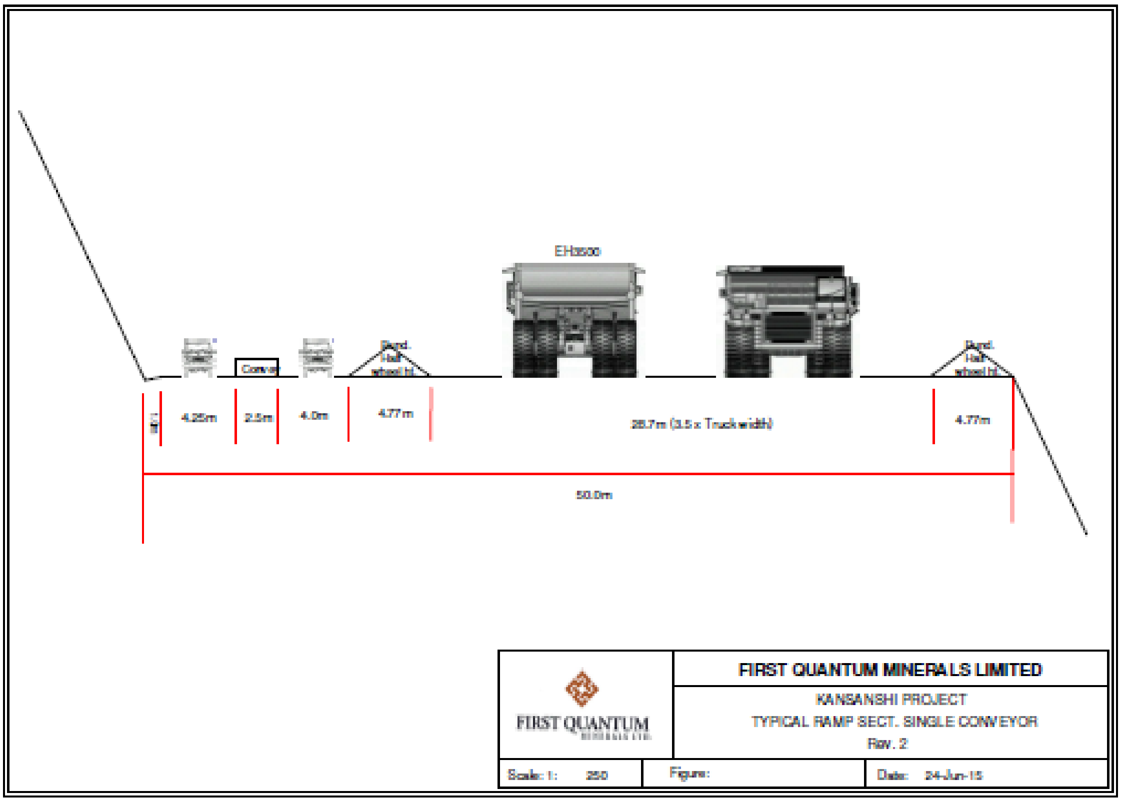
16.4.1 Mine design parameters

Basic mine design parameters relating to pit slope parameters 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 = 28.7 m
- haul road width with single conveyor, inclusive of bunds, conveyor access corridors and side drain = 50 m (refer to schematic in Figure 16-6)
- maximum gradient to allow crusher tramming = 1 : 10

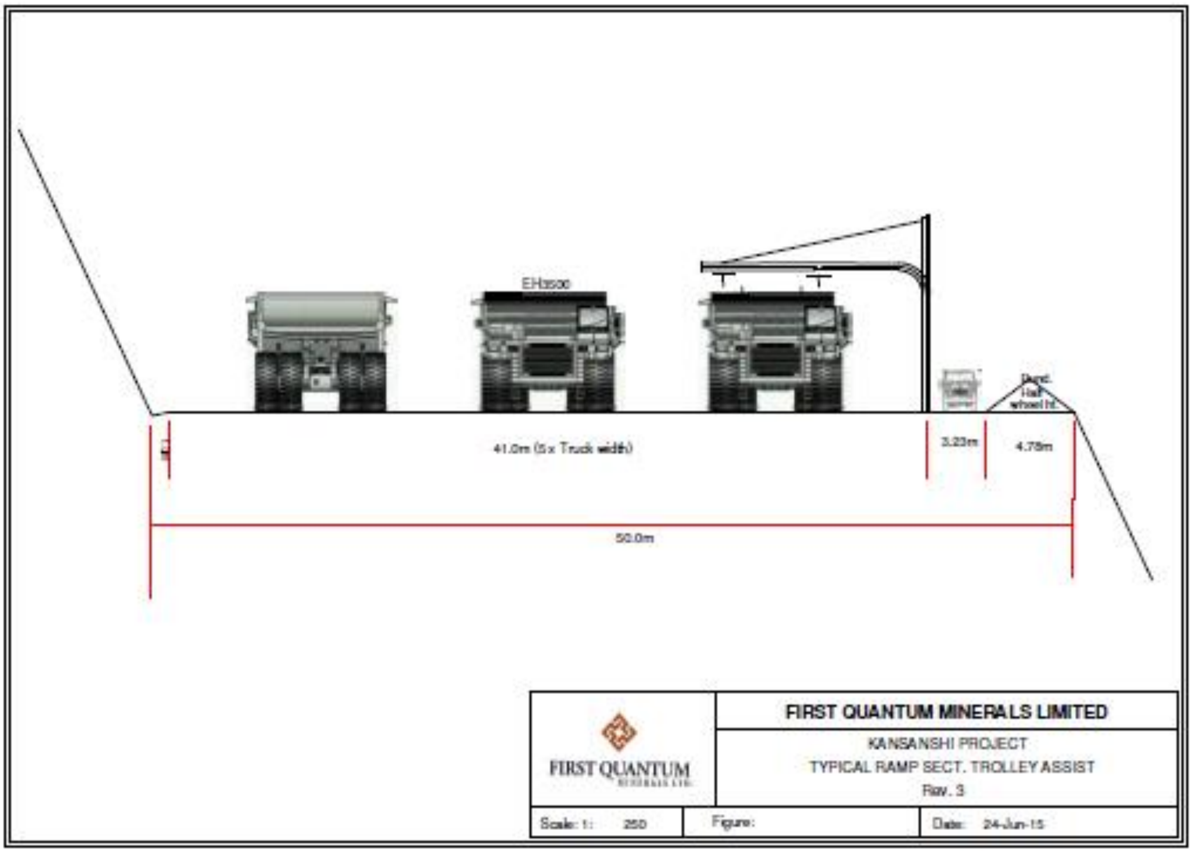
Figure 16-6 Schematic cross section across haul road including conveyor



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

- haul road minimum width = 5 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 16-7)
- maximum gradient = 1 : 10

Figure 16-7 Schematic cross section across haul road for trolley-assist haulage



Further detailed parameters have had to be considered for IPCC implementation. ThyssenKrupp engineering drawings showed the following required excavation and installation dimensions for each semi-mobile crusher installation:

- excavated height of crusher pocket = 19.4 m
- dimensions (area) at base of crusher pocket = 18.0 m wide x 25.5 m deep
- pocket batter angle = 75°
- bin capacity above crusher = 300 m^3
- bin capacity below crusher and above transfer conveyor = 550 t
- length of draw- bridges = 12.98 m
- width of draw-bridges = 14.92 m at opening, closing to 11.8 m at tip-head
- clearance to under-side of overhead crane rails = 10.8 m

In relation to the conveyor specifications, the following parameters were considered:

- length of feeder conveyor (ie, between the crusher and discharge conveyor) = 42.5 m minimum
- nominal minimum length of discharge conveyor (ie, between transfer points) = 1,000 m
- width of discharge conveyor belt = 1.6 m belt width; 2.5 m structural width
- minimum width of service corridor alongside discharge conveyor = 4.0 m
- feeder conveyor must be positioned at 90° to discharge conveyor

Additional IPCC design considerations include:

- Installations are to be established near-surface initially, but must proceed to deeper positions or to new locations in other pits, 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.
- Conveyor belt transfer point(s) between the pit crests and the plant are fixed.
- 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 pit cutbacks and ensure separation.
- Where possible, design for long, straight discharge conveyor routes.
- Minimise haul ramp / conveyor belt crossovers. Minimise in-pit transfer points.

The following design parameters relate to the accommodation of trolley-assist haulage routes in the pit designs:

- The trolley-assist system will require triple lane haul ramps (up, down and drop-off lanes) for efficient operation and have been designed to be 50 m wide at 1:10 gradient.
- Non trolley-assist ramps have also been designed at 1:10 gradient and for those ramps that do not include IPC conveyors, the design width is 36 m. The design width of those ramps including IPC conveyors is 45 m.

16.4.2 **Mine geotechnical engineering**

KMP geotechnical engineers have produced a detailed table of pit slope design parameters, based on experience within the current operations and in exposed batters and overall slopes throughout the mine. Table 16-2 shows these detailed parameters, whilst Figure 16-8 shows how these parameters relate to design sectors around the Main and North West Pits.

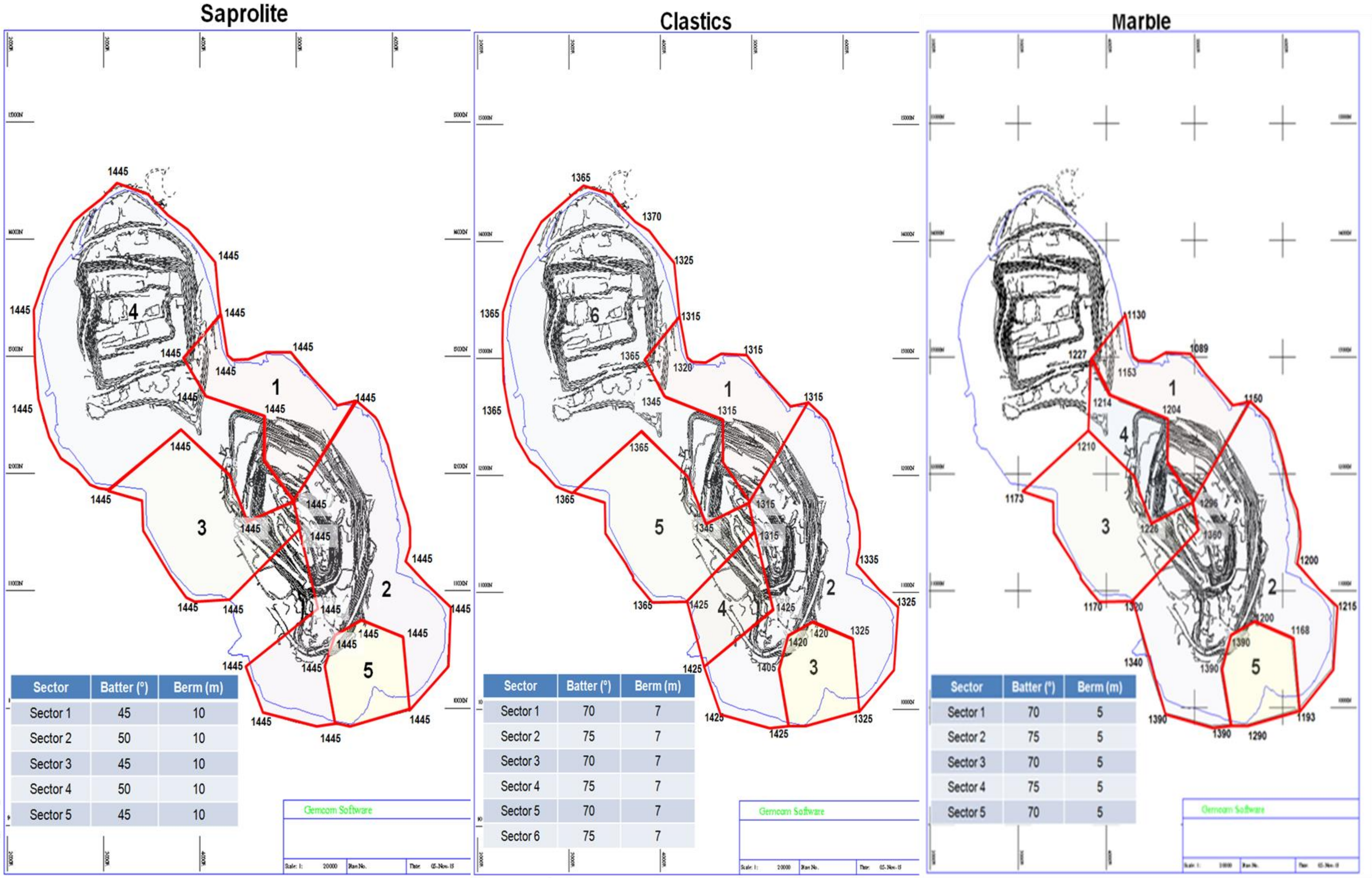
In practice, these parameters were too complex to apply in the design process. For design purposes, the parameters needed to be rationalised into the three main zones of saprock, clastics and marble. The specified batter angles in each of the three zones varied by 5^{o}. The flattest of the angles was adopted for the saprock and the marbles, whilst the steeper of the angles was adopted for clastics in the deeper portion of the pit. This was because of the number of ramps on each wall in this deep conical zone of the pit, having the effect of flattening the overall wall angle.

Design parameters within the three similar domains for South East Dome were adopted from the same rationalisation.

Table 16-2 Pit slope design parameters

Pit	Domain	Sector	Zones	Applicable Elevations*	Inter-ramp Slope		Batter Angle	Bench Height	Berm Width	Ramp Width	Comments
					Crest to Crest	Toe to Crest					
Main	Saprolite	1	M5 north, M9B, M10 north,	1445 - 1315	26.5	33.7	45	10	10	30	Max stack height of 50m. Double berm required
		2	M5 east, M6, M4 east, M8B south	1445 - 1335 (min) to 1315(max)	28.5	36.7	50	10	10	30 (50 for conveyor)	
		3	M7 north, M8A north, M8B north, M10 south	1445 - 1365 (min) to 1315 (max)	26.5	33.7	45	10	10	30 (50 for conveyor)	Max stack height of 50m. Double berm required
		4	M10 central, M11, M9A	1445 - 1365 (min) to 1315 (max)	28.5	36.7	50	10	10	30	
		5	M6 Southern Extension	1445 - 1425 (min) to 1325 (max)	26.5	33.7	45	10	10	30	Bullnose
	Clastics	1	M5 north, M9B, M10 north,	1315 - 1266 (min) to 1089 (max)	43.2	54.5	70	10	7	30	
		2	M5 east, M6 east, M4 east	1425 (min) to1315 (max) - 1340 (min) to 1150 (max)	45.9	58.3	75	10	7	30 (50 for conveyor)	
		3	M6 Southern Extension	1425 - 1325 (min) to 1390 - 1168 (max)	43.2	54.5	70	10	7	30	Bullnose
		4	M8A south, M6 South	1425 - 1320	45.9	58.3	75	10	7	30 (50 for conveyor)	
		5	M7 north, M8A north, M10 south	1365 (min) to 1315 (max) - 1380 (min) to 1162 (max)	43.2	54.5	70	10	7	30 (50 for conveyor)	
		6	M9A, M11, M10 central	1365 (min) to 1315 (max) - 1296 (min) to 1204 (max)	45.9	58.3	75	10	7	30	
	Marble	1	M5 north, M9B, M10 north,	1296 (min) to 1089 (max) - 1018	49.2	58.5	70	10	5	30	
		2	M5 east, M6, M4 east, M8A south, M8B south	1380 (min) to 1150 (max) - 1018	52.5	62.6	75	10	5	30 (50 for conveyor)	
		3	M7 north, M8A north, M10 south	1380 (min) to 1162 (max) - 1018	49.2	58.5	70	10	5	30 (50 for conveyor)	
		4	M9A, M11, M10 central	1296 (min) to 1204 (max) - 1018	52.5	62.6	75	10	5	30	
		5	M6 Southern Extension	1390 to 1168 - 1018	49.2	58.5	70	10	5	30	Bullnose
North West	Saprolite	1	S4	1445 - 1385 (min) to 1315 (max)	26.5	33.7	45	10	10	30	
		3	S4	1445 - 1365 (min) to 1315 (max)	26.5	33.7	45	10	10	30	
		4	NW pit	1445 - 1365 (min) to 1315 (max)	28.5	36.7	50	10	10	30 (50 for conveyor)	
	Clastics	1	S4	1365 (min) to 1315 (max) - 1227 (min) to 1130 (max)	43.2	54.5	70	10	7	30	
		5	S4	1365 - 1210 (min) to 1163 (max)	43.2	54.5	70	10	7	30	
		6	S3, S4	1365 - 1130	45.9	58.3	75	10	7	30 (50 for conveyor)	
	Marble	1	S4	1227 (min) to 1130 (max) - 1018	49.2	58.5	70	10	5	30	
		4	S3, S4	1227 (min) to 1210 (max) - 1018	52.5	62.6	75	10	5	30 (50 for conveyor)	

Figure 16-8 Pit slope design sectors



Saprolite

Sector	Batter (°)	Berm (m)
Sector 1	45	10
Sector 2	50	10
Sector 3	45	10
Sector 4	50	10
Sector 5	45	10

Clastics

Sector	Batter (°)	Berm (m)
Sector 1	70	7
Sector 2	75	7
Sector 3	70	7
Sector 4	75	7
Sector 5	70	7
Sector 6	75	7

Marble

Sector	Batter (°)	Berm (m)
Sector 1	70	5
Sector 2	75	5
Sector 3	70	5
Sector 4	75	5
Sector 5	70	5

To enable this rationalisation to be applied, a geotechnical block model was produced, coded with lithology (Figure 16-9) and with the following block attributes and geotechnical specifications:

- 1 = Saprolite / Saprock; batter Angle = 45°; berm width = 10 m; bench height = 10 m
- 21 to 24 = Clastics; batter angle = 75°; berm width = 7 m; bench height = 10 m
- 31 to 34 = Marbles; batter angle = 70°; berm width = 5 m; bench height = 10 m

Figure 16-9 Geotechnical block model coded with lithology

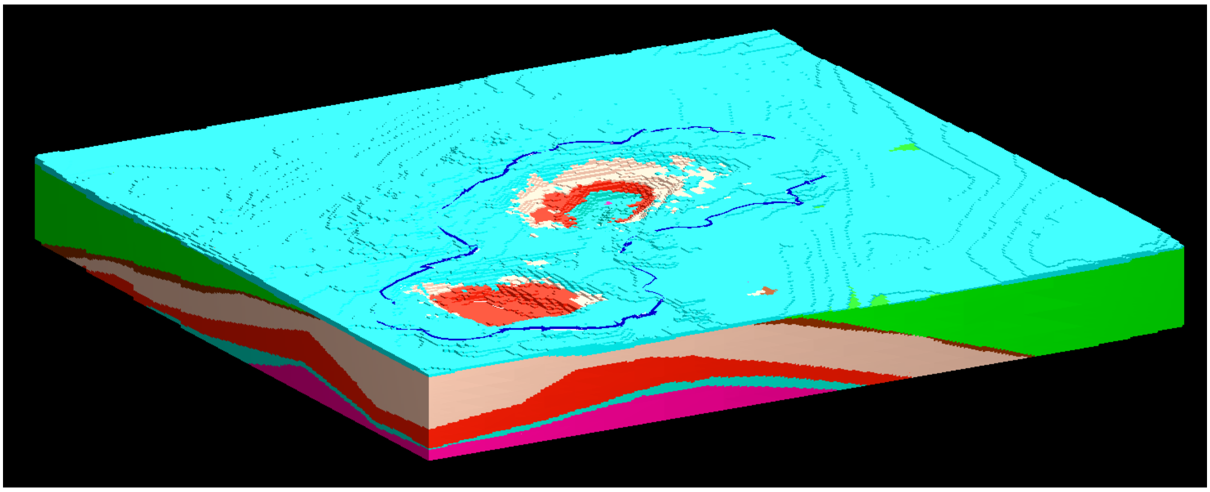
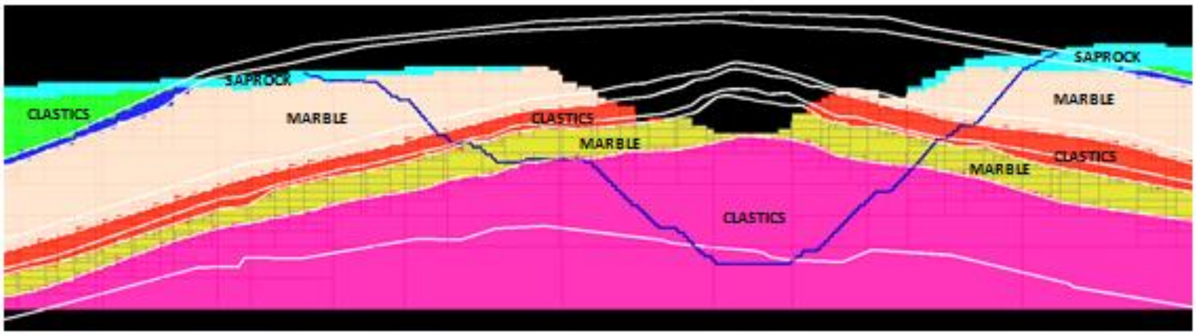


Figure 16-10 shows an example cross section through the coded model for Main Pit.

Figure 16-10 Example cross - section through deepest part of Main Pit



16.4.3 **Mine dewatering**

Dewatering of the open pits is currently carried out by means of collection and pumping from in-pit sumps. Dewatering was achieved previously from perimeter and in-pit boreholes and from the old Kansanshi underground mine shaft.

A decline from the eastern wall of the Main Pit has been developed as part of a longer-term strategy for mine dewatering and is contributing to the majority of abstraction volume. The decline has been developed as a series of 4 m x 4 m dimensioned straights descending to elevations below the long-term pit floors. From these positions, pump stations have been constructed. Water will be pumped along the decline to a 4.1 m diameter bored raise (vertical), and thence to surface and into a storage dam to supplement the process water demand.

Records from monitoring bores and observations of "drying-up" of pit seepage zones, indicate that the decline and shaft development has been effective in lowering the groundwater surface below the level of the current pit push-backs in Main Pit.

16.4.4 **Mining dilution and recovery**

Production tracking and reconciliation records exist for the mining operations and a review has been made of these records for 2014. It is possible to track the mining dilution and mining recovery from the grade control model blocks (and grades), through the marked-out dig block attributes (ie, planned mining) and to the dispatched tonnes and grades (ie, actual mining).

The figures vary from month to month but overall, the records for 2014 indicate an average mining dilution of approximately 110% and a mining recovery of approximately 98%. However, there are inaccuracies in the recording of this information due to:

- material movements unaccounted for (ie, untracked contractor mining, inclusion of sheeting, windrows and ramp volumes)
- no survey volume adjustments (ie, swell factor variance)
- dispatched blocks assigned in-situ densities rather than swelled bulk densities (ie, leading to tonnage variance)

On this basis, it is considered prudent to maintain the mining dilution and mining recovery factors adopted in the 2012 Technical Report (Journet and Cameron, DumpSolver, 2012).

16.5 **Mining and processing schedule**

16.5.1 **LOM schedule**

Features of the LOM mining and production schedule as listed in Table 16-3 are as follows:

- The production shown for 2015 represents that for the remaining seven months of the year. From 2016 the remaining Project life is 16 years (ie, to 2031).
- The total material mined from all pits amounts to 2,119.2 Mt (845.2 Mbcm), of which 668.0 Mt is ore (including mixed-leach, mixed-float and sulphide ore) and 1,451.2 Mt is waste (including refractory and inferred ore). 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%.
- The direct feed float-leach ore is 134.5 Mt at a grade of 1.18%Cu and 46.4 Mt at a grade of 0.55% Cu is stockpile reclaim from existing stockpiles.
- The total float-leach ore mined to stockpiles and reclaimed throughout the Project life is 70.3 Mt at a grade of 0.60% Cu.
- The direct feed sulphide ore is 354.8 Mt at a grade of 0.67%Cu and 7.2 Mt at a grade of 0.36% Cu is stockpile reclaim from existing stockpiles.
- The total sulphide ore mined to stockpiles and reclaimed throughout the Project life is 108.4 Mt at a grade of 0.30% Cu.
- The current plant configuration has a capacity of 15.6 Mtpa of float-leach feed and 13.2 Mtpa of sulphide feed. In mid-2017 as a first phase of the S3 expansion, the total sulphide capacity is increased by 12.4 Mtpa. In 2020, upon completion of the S3 expansion, the sulphide capacity is increased a further 12.6 Mtpa to reach a maximum of 38.2 Mtpa.

- The average copper grade for float-leach feed between 2015 and 2019 is 1.20% Cu. The average copper grade for sulphide feed during the same period is 0.88% Cu. Thereafter the average feed grades drop to 0.86% Cu in float-leach and 0.57% Cu in sulphide over the rest of the LOM period.
- Current annual copper metal production is 250,000 tonnes. Between 2017 and 2027, the average annual copper metal production is 305,000 tonnes (on a diluted feed basis), before tailing off in the final years of production.
- The annual average gold production is approximately 104 thousand ounces (on a diluted feed basis).
- The overall life of mine strip ratio (tonnes) is 2.16 : 1.
- Main Pit is mined extensively throughout the life of the mine and has three production areas. A central area, which is already stripped of waste and ore, is exposed and currently being mined. The other areas are phase 1 push back areas which are the next areas designated for pre-strip mining, and the ultimate pushbacks to achieve the final pit configuration.
- The North West Pit is depleted by 2019 in the schedule and thereafter is available for in-pit backfill of waste from the Main Pit pushback areas.
- The South East Dome Pit is mined from 2017 to 2023 at an average annual mining rate of around 40 Mt.
- The current stockpiles are rehandled between 2016 and 2027 at an average rate of around 5 Mtpa. The maximum float-leach stockpile size of 81.6 Mt is reached in 2022, whereas the maximum sulphide stockpile of size 76.4 Mt is reached in 2023.

Figure 16-11 to Figure 16-13 depict the LOM schedule graphical results.

16.5.2 **Waste dumping schedule**

Table 16-4 provides a waste dumping schedule associated with the LOM schedule listed in Table 16-3. Aspects of this schedule include:

- 497.7 Mlcm can be hauled onto the Congo dump
- 263.6 Mlcm can be backfilled into the depleted North West Pit

The total volume dumped is 761.4 Mlcm, which is 585.7 Mbcm assuming a 30% swell factor. This equates to the total volume mined in the LOM production schedule.

Table 16-3 Kansanshi Operations life of mine production schedule

	UNITS		TOTAL	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
MINING																				
Total ore (incl. mining dil'n/rec.)	Tonnes	Mt	**668.0**	30.7	60.2	41.5	63.3	56.9	51.9	53.3	58.5	61.6	36.9	33.9	45.8	34.6	23.0	10.7	4.1	0.9
	Cu	%	**0.71**	0.88	0.88	0.82	0.78	0.76	0.74	0.71	0.73	0.62	0.68	0.61	0.50	0.54	0.55	0.46	0.56	0.69
Total waste (incl. mining dil'n/rec.)	Tonnes	Mt	**1,451.2**	82.1	97.7	130.1	130.6	120.4	143.0	141.6	128.3	133.3	93.5	69.0	84.9	50.8	29.4	13.0	3.1	0.4
Strip ratio	W : O	t: t	**2.17**	2.67	1.62	3.14	2.06	2.12	2.75	2.65	2.19	2.16	2.53	2.04	1.85	1.47	1.28	1.21	0.75	0.43
Direct feed (incl. mining dil'n/rec.)	Tonnes	Mt	**489.3**	15.9	26.60	28.94	33.83	35.74	46.22	45.24	49.06	42.34	25.97	31.49	38.44	33.43	20.87	10.65	3.72	0.86
	Cu	%	**0.81**	1.14	1.00	1.06	1.09	1.05	0.80	0.79	0.81	0.77	0.86	0.63	0.55	0.55	0.58	0.47	0.59	0.73
Stockpile on (incl. mining dil'n/rec.)	Tonnes	Mt	**178.8**	14.9	33.59	12.54	29.50	21.14	5.71	8.09	9.48	19.28	10.94	2.38	7.37	1.22	2.12	0.09	0.36	0.08
	Cu	%	**0.43**	0.61	0.78	0.26	0.43	0.27	0.27	0.28	0.33	0.29	0.26	0.32	0.23	0.34	0.27	0.23	0.23	0.23
Stockpile off (incl. mining dil'n/rec.)	Tonnes	Mt	**232.4**	0.0	2.13	5.98	7.26	5.36	7.45	8.42	4.60	11.32	27.70	22.18	15.23	20.23	32.79	37.11	17.01	7.60
	Cu	%	**0.44**	0.00	1.21	0.66	0.46	0.46	0.54	0.53	0.50	0.50	0.45	0.37	0.52	0.47	0.34	0.35	0.54	0.28
Stockpile Balance	Tonnes	Mt		68.5	99.9	106.5	128.7	144.5	142.8	142.5	147.3	155.3	138.5	118.8	110.9	91.9	61.2	24.2	7.5	0.0
	Cu	%		0.54	0.61	0.57	0.54	0.50	0.49	0.48	0.47	0.44	0.43	0.44	0.41	0.40	0.43	0.54	0.54	0.00
TOTAL FEED TO PLANT (after mining dil'n & recovery)																				
Mixed float/leach	Tonnes	Mt	**251.2**	9.9	15.6	15.6	15.6	15.6	15.6	15.6	15.6	15.6	15.6	15.6	15.6	15.6	15.6	15.6	15.6	7.9
	Cu	%	**0.89**	1.27	1.15	1.18	1.21	1.20	0.95	0.89	1.02	0.86	0.94	0.80	0.76	0.68	0.63	0.59	0.60	0.30
	Au	g/t	**0.12**	0.14	0.15	0.12	0.16	0.09	0.12	0.10	0.11	0.12	0.12	0.13	0.12	0.13	0.12	0.13	0.13	0.13
Sulphide float	Tonnes	Mt	**470.4**	6.0	13.2	19.4	25.5	25.5	38.1	38.1	38.1	38.1	38.1	38.1	38.1	38.1	38.1	32.2	5.2	0.5
	Cu	%	**0.58**	0.91	0.87	0.84	0.84	0.83	0.69	0.69	0.69	0.65	0.53	0.41	0.45	0.45	0.35	0.27	0.38	0.59
	Au	g/t	**0.12**	0.09	0.13	0.13	0.16	0.13	0.12	0.12	0.12	0.10	0.11	0.10	0.09	0.11	0.11	0.12	0.12	0.24
Total feed	**Tonnes**	**Mt**	**721.6**	**15.9**	**28.7**	**34.9**	**41.1**	**41.1**	**53.7**	**53.7**	**53.7**	**53.7**	**53.7**	**53.7**	**53.7**	**53.7**	**53.7**	**47.8**	**20.7**	**8.5**
	Cu	**%**	**0.69**	**1.14**	**1.02**	**0.99**	**0.98**	**0.97**	**0.76**	**0.75**	**0.78**	**0.71**	**0.65**	**0.52**	**0.54**	**0.52**	**0.43**	**0.38**	**0.55**	**0.32**
	Au	**g/t**	**0.12**	**0.12**	**0.14**	**0.13**	**0.16**	**0.11**	**0.12**	**0.11**	**0.12**	**0.10**	**0.11**	**0.11**	**0.10**	**0.12**	**0.11**	**0.13**	**0.13**	**0.14**
AVERAGE RECOVERIES																				
Mixed float/leach	Cu	%	**77.9%**	79.8%	79.3%	79.0%	78.0%	79.5%	77.4%	77.3%	78.4%	78.4%	78.0%	76.9%	77.5%	79.5%	62.7%	79.8%	79.7%	79.7%
	Au	%	**66.2%**	65.3%	65.6%	65.5%	65.9%	65.9%	66.3%	66.3%	66.2%	66.4%	66.4%	66.5%	66.5%	66.3%	66.6%	66.4%	66.4%	67.4%
Sulphide float	Cu	%	**89.4%**	87.9%	88.5%	89.4%	86.6%	88.6%	90.1%	89.9%	89.8%	89.8%	88.9%	89.2%	90.1%	90.5%	90.0%	89.8%	89.5%	88.4%
	Au	%	**63.4%**	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%
Overal average	Cu	%	**84.1%**	**82.4%**	**83.3%**	**83.8%**	**82.4%**	**84.1%**	**85.0%**	**85.5%**	**85.9%**	**85.5%**	**84.3%**	**83.4%**	**84.8%**	**86.8%**	**78.2%**	**84.8%**	**81.3%**	**80.8%**
	Au	%	**64.4%**	**64.8%**	**64.6%**	**64.3%**	**64.4%**	**64.2%**	**64.3%**	**64.1%**	**64.2%**	**64.4%**	**64.4%**	**64.5%**	**64.5%**	**64.4%**	**64.4%**	**64.4%**	**65.7%**	**67.0%**
METAL RECOVERED (after mining dil'n & recovery)																				
Mixed float/leach	Cu	kt	**1,749.8**	100.4	142.4	144.3	147.6	149.0	114.2	107.7	124.7	104.9	114.3	95.9	91.5	84.4	61.7	73.3	74.3	19.2
	Au	koz	**666.5**	28.7	47.9	40.0	51.5	29.9	40.5	32.5	37.4	39.0	41.1	43.1	40.0	43.5	41.2	44.1	43.1	23.0
Sulphide float	Cu	kt	**2,443.9**	48.3	101.5	145.9	184.2	187.3	234.6	236.1	236.7	221.7	178.8	138.9	154.6	156.5	119.6	79.0	17.6	2.8
	Au	koz	**1,105.7**	11.5	35.1	52.0	81.2	66.7	93.8	93.5	92.2	75.8	86.2	79.0	72.7	86.9	84.3	79.9	12.2	2.7
Total recovered	Cu	kt	**4,193.7**	148.8	243.8	290.2	331.8	336.3	348.8	343.7	361.4	326.6	293.0	234.8	246.0	240.9	181.3	152.3	91.9	22.1
	Au	koz	**1,772.2**	40.2	83.1	92.0	132.7	96.5	134.3	126.0	129.6	114.8	127.3	122.1	112.6	130.4	125.5	124.0	55.3	25.7

Figure 16-11 LOM scheduling – annual mining materials movement

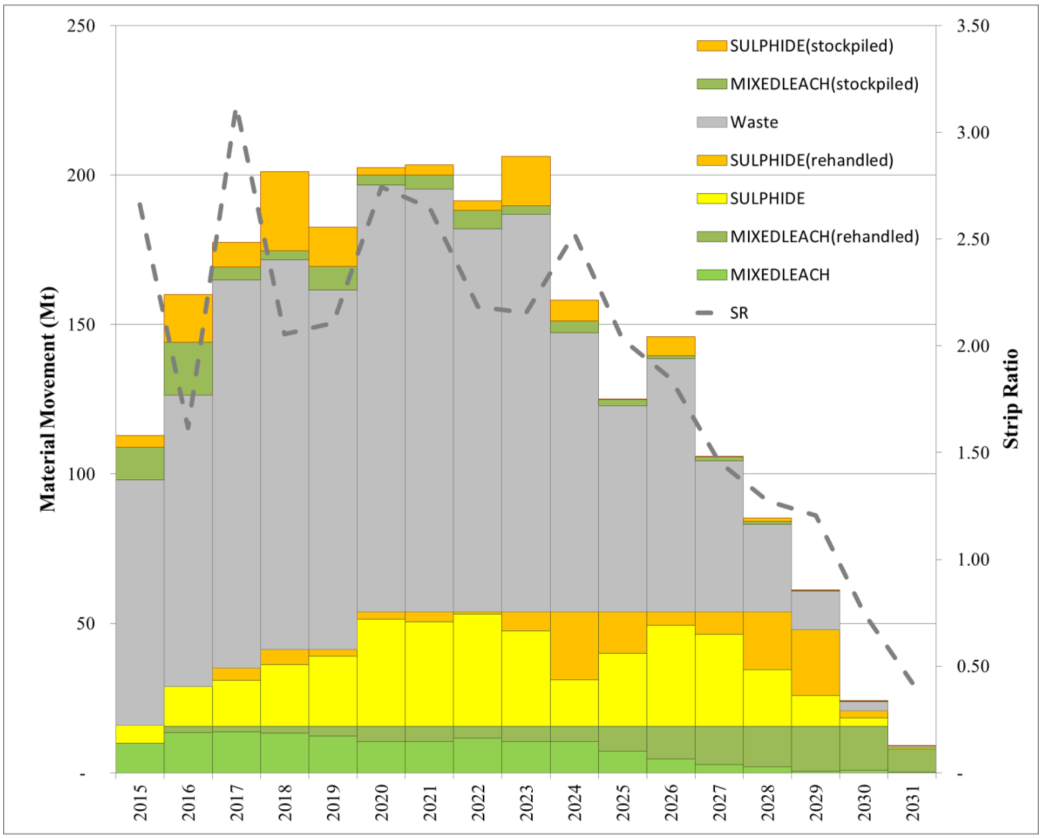


Figure 16-12 LOM scheduling – mining sequence

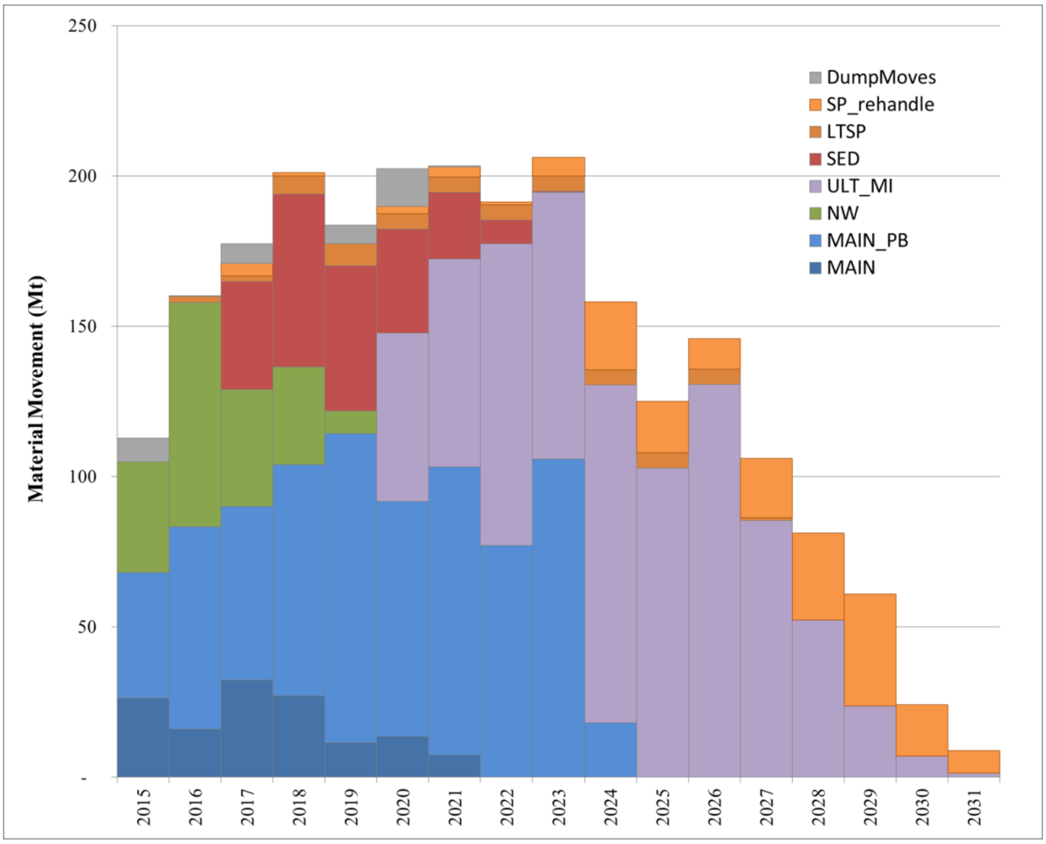


Figure 16-13 LOM scheduling – annual copper ore production

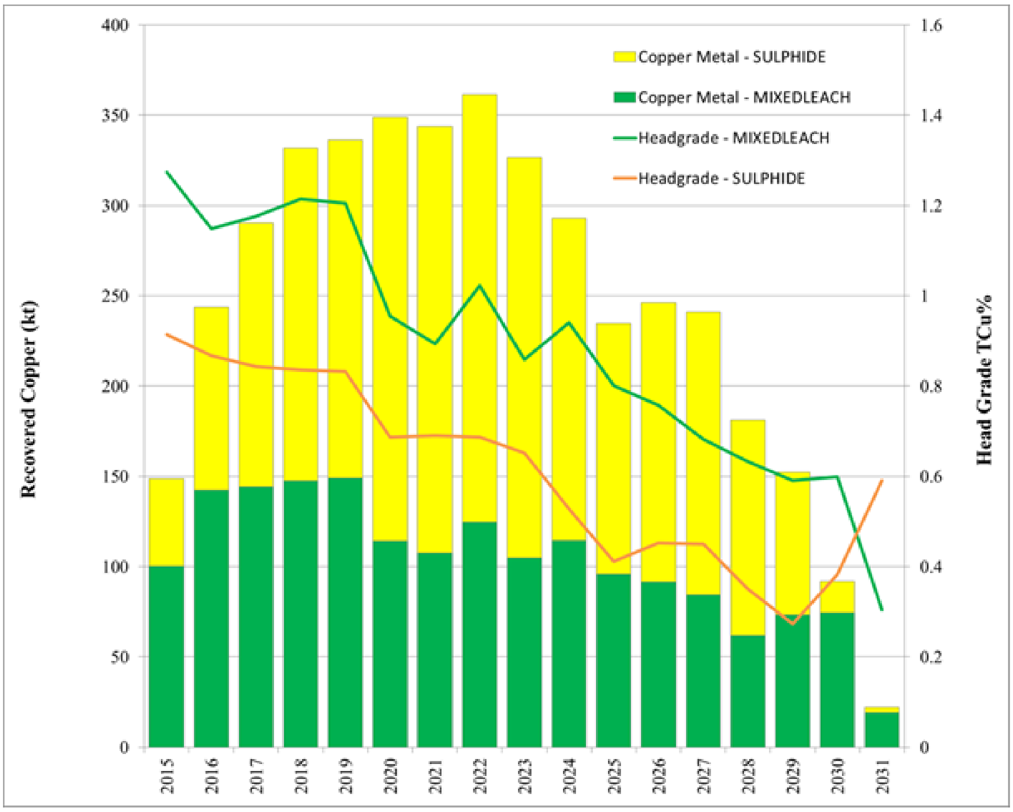


Table 16-4 Kansanshi Operations life of mine production schedule

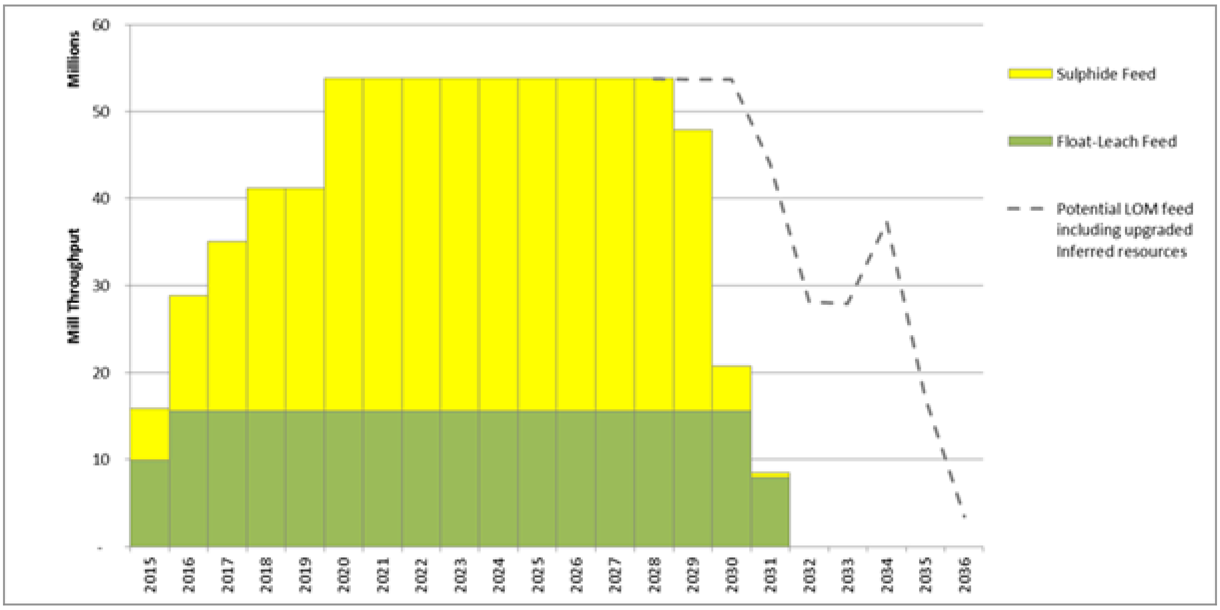
	Total	**2015**	**2016**	**2017**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**	**2030**	**2031**
To dump (Mbcm)	**497.7**	49.5	51.2	75.3	65.7	48.3	65.0	43.9	33.5	41.6	23.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0
To NW Pit backfill (Mbcm)	**263.6**					19.9	17.8	34.2	27.9	24.0	21.5	33.0	40.0	23.9	13.8	6.1	1.4	0.2
Total	**761.4**	49.5	51.2	75.3	65.7	68.2	82.8	78.0	61.4	65.6	45.2	33.0	40.0	23.9	13.8	6.1	1.4	0.2

16.5.3 **LOM schedule inclusive of Inferred Mineral Resource**

As advised in Item 15.5.3, M+I+I designs allow for production planning, planning of ultimate waste dumping extents, and for the planning of infrastructure requirements. Figure 16-14 shows a notional plant feed graph for a LOM scheduling scenario associated with the M+I+I pit designs and including as additional plant feed, currently Inferred Mineral Resource (above marginal cut-off grade). The inclusion of this feed is on the proviso that this resource can be upgraded to at least an Indicated classification.

Figure 16-14 shows that an upgrade of Inferred Mineral Resources has the potential to feed the mills at full S3 plant capacities until 2031 before tailing off gradually to end in 2036. The notional profile shonw here is for information only and the related inventory forms no part of the Mineral Reserves estimation process.

Figure 16-14 LOM scheduling – plant feed profile with the inclusion of currently Inferred Mineral Resource



ITEM 17 **RECOVERY METHODS**

The following information is reproduced in part from the December 2012 Technical Report (Journet and Cameron, DumpSolver, 2012), with an update provided by Robert Stone (QP).

17.1 **Current processing facilities**

The current Kansanshi flowsheet is shown in Figure 17-1.

Mixed float and mixed leach ores are treated via crushing, milling, flotation, sulphuric acid leaching and the SXEW process to produce cathode copper. The ore is treated with a sulphidising agent, NaHS (sodium hydrosulphide) to assist with flotation to recover a proportion of acid-insoluble copper minerals and gold. The concentrate so produced is filtered (pressure filters) and sold along with concentrate from the separate sulphide and mixed ore processing plants.

Sulphide ore is treated via crushing, milling and flotation to produce copper in concentrate. High pressure leach (HPL) autoclaves are used to treat a portion of the concentrate product and produce a supplementary oxidised feed suitable for SXEW cathode copper recovery. A fourth electro winning facility was commissioned in Q3 2008, and, alongside a third SX train, provides the capacity to handle the additional copper input from the sulphide HPL circuit.

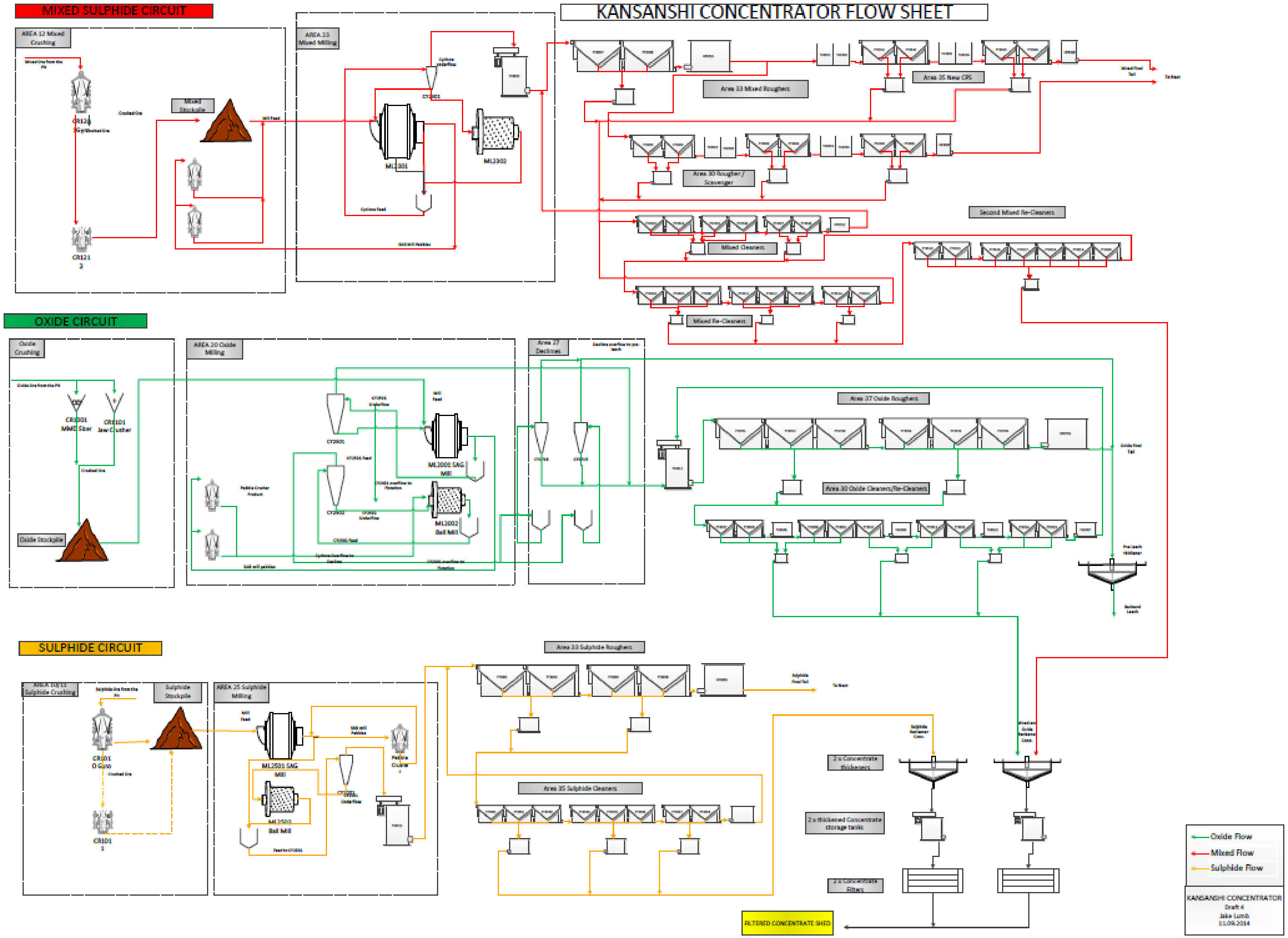
In 2009, the HPL plant was treating FQM's Frontier concentrate on a toll treatment basis. This avoided the loss of payable gold in the concentrate treated. After the closure of the Frontier operations, HPL treatment of Kansanshi concentrate was resumed. At that time, metallurgical testwork indicated that gravity gold recovery was possible on HPL residues and an acid resistant gravity concentrator was installed.

Gold recovery by gravity in the main milling plants was expanded by the addition of four new gravity concentrators in April 2010, thus providing two concentrators per milling train, and increasing gold recovery from all ore types. Shaking tables were also installed to treat gravity gold concentrates with the final concentrate being smelted to produce dore.

The 2002 DFS (GRD MInproc, 2002) and preliminary studies undertaken on behalf of FQM envisaged a logical staged development of milling facilities at Kansanshi to coincide with the ore mining schedule. The initial plant proposed annual treatment rates of 4 million tonnes of oxide ore and 2 million tonnes of sulphide ore (4O+2S). Infill drilling determined that near surface mining would encounter more sulphide material than first considered. During construction, capital additions were made to the sulphide milling circuit to double the treatment capacity of sulphide ore to 4 million tonnes (4O+4S). Open pit mining commenced in 2004.

In early 2006, a second expansion programme was completed to increase sulphide treatment capacity to 8 million tonnes (4O+8S). Also in 2006, an HPL facility was completed to treat the increased copper concentrate output. Actual throughput of the HPL facility is carefully managed and depends on factors such as third party smelter capacity, electrowinning capacity and power availability.

Figure 17-1 Current Kansanshi Flowsheet



A third expansion of the sulphide circuit was completed in 2008, thereby increasing capacity to 12 Mtpa.

Up until 2009 mixed ores were not treated as run of mine and were stockpiled. In early 2009, metallurgical test-work indicated that mixed ores could be successfully treated and a 4 million tonne mixed ore circuit was added during 2009. With the series of upgrades to the sulphide circuit, the treatment configuration at Kansanshi became (4O+4M+12S).

Also in 2009, Kansanshi completed the addition of gravity concentrators and upgrades to the gold plant and as a consequence of these additions, gold production was boosted in 2009 to 100,000 ounces. Further debottlenecking of the three circuits, coupled with switching of the mill circuits to suit desired ore type treatment rates, has allowed variable ratios of sulphide, mixed and oxide ore tonnages to be treated. The combined circuits have a capacity range from, typically, 7.3O+6.6S+15.3M to 7.3O+9.3M+13.2S. With the expansion of the leach circuit noted below in section 17.2, the latter becomes the normal operating scenario.

17.2 **Leach expansion project**

The current leach processing facilities have been expanded to 16.6 Mtpa capacity, largely incorporating a continuation of the existing leach/CCD/SX and EW technology. This expansion involves the installation of an ore sizer for initial comminution, and the milling of leachable ore through the existing oxide mills and the original sulphide milling circuit (previously being used for mixed ore treatment).

17.3 **S3 sulphide expansion project**

The Kansanshi S3 sulphide expansion project is an expansion to the existing Kansanshi processing facilities. The expansion includes the construction of a copper concentrator capable of treating an additional 25 Mtpa of sulphide ore, and an overland conveyor route from a near-mine surface transfer bin receiving crushed ore from the in-pit crushers. Refer to Figure 15-8 for the location.

The proposed plant design is based on well understood and proven technology. The flow sheet has been established based on the operations of the existing sulphide copper ore treatment circuits installed in 2004 and 2006 which have been operating consistently and efficiently since these times. The design of S3 is based on the design of one half of the Trident operation.

Details on specific flowsheet development and specific process design aspects are summarised below from a stand-alone report prepared by Lycopodium (September, 2012).

17.3.1 **Flowsheet**

The mechanical design of the S3 plant is based on the principles of the design developed for the Company's Trident project with S3 utilising one milling circuit rather than the two of Trident. This allows commonality of design and equipment. The S3 metallurgical design is based upon the current Kansanshi sulphide circuit as this has been proven on the ore types to be treated.

17.3.2 Plant throughput

A plant throughput rate of 25 Mtpa was nominated for the S3 design. Based on 8,000 operating hours per year, this is equivalent to a mill feed rate of 3,125 tph and at the nominal head grade of 0.66% Cu and 95% recovery, equates to 653,125 tpa copper concentrate production at 24% concentrate grade.

17.3.3 Process design parameters

The key process design parameters adopted by Lycopodium were as listed in Table 17-1.

Table 17-1 S3 Sulphide Plant, Process Design Criteria

Parameter	Value	Source
Annual Throughput	25 Mtpa	FQM
Copper Head Grade (Nominal)	0.66%	FQM
Gold Head Grade	0.15 g/t	FQM
Concentrate Copper Grade (Nominal)	24%	FQM
Copper Recovery	95%	FQM
Crushing Plant Availability	4,672 h/annum (80%)	Industry
Concentrator Availability	8,000 h/annum (91.3%)	FQM
Crusher Operation	16 h/day	FQM
Primary Crusher (Operating)	5,351 dtph	Calculated
Mill Circuit Feed Rate	3,125 dtph	Calculated
SAG Mill Feed to Cyclone Overflow	20% of New Feed	Agreed with FQM
Cyclone Underflow Split to SAG Mill	20% of New Feed	Agreed with FQM
Ball Mill Circulating Load (Design)	350%	Lycopodium
Flow Rate to Gravity Circuit	30% of Ball Mill Circulating Load	FQM
Rougher Scavenger Residence Time (Lab)	6 min	Confirmed by KMP
Cleaner Residence Time (Lab)	3 min	Confirmed by KMP
Cleaner Scavenger Residence Time (Lab)	3 min	Confirmed by KMP
Recleaner Residence Time (Lab)	3 min	Confirmed by KMP
Tails Thickener Settling Flux	0.76 $t/m^2.h$	Confirmed by KMP
Concentrate Thickener Settling Flux	0.10 $t/m^2.h$	Confirmed by KMP
Concentrate Filtration Rate (Design)	0.5 $t/m^2.h$	Confirmed by KMP
Final Concentrate Make (Cu equivalent)	653,125 dtpa	Calculated

17.3.4 Primary crushing and crushed ore storage

Primary crushing will be carried out by two semi-mobile, in-pit, independent gyratory crushers (IPC) operating in open circuit. Two crushers will be installed ahead of the first year plant production, at timeframes advised in Item 16.1.6.

Each crusher will be equipped with a transfer conveyor of 42.5 m fixed length, and a variable length discharge conveyor connecting the IPC facility to a surface transfer point (Figure 15.8 and Figure 16.1). The mine plan attempts to minimise the number of in-pit transfer points along the discharge conveyor route. At the pit top transfer point, IPC product will be transferred onto an overland conveyor and thence overland to the S3 plant stockpile.

Crusher operation will be for nominally 16 hours each / day with a lower availability of 85% requiring an average feed rate to the crushed ore stockpile of 5,351 tph. The crushed ore stockpile will have 13 hours live capacity, equivalent to about 40,000 tonnes and be fed by conveyor. Ore will be recovered from below the crushed ore stockpile by a four variable speed apron feeders located in a single reclaim tunnel.

17.3.5 **Grinding circuit**

Crushed ore reclaimed from the stockpile will be fed to a semi-autogenous grinding (SAG) mill. The 28 MW mill will be 12.2 m in diameter with an effective grinding length (EGL) of 8.2 m. A high-low liner configuration has been proposed, without pebble ports in the discharge grate. Oversize material from the SAG mill discharge screen will fall onto a conveyor and be transferred back to the SAG mill feed conveyor. Allowance has been made for future crushing of the oversize should this be required. The SAG mill will discharge into a sump and pumped to a dedicated cyclone pack.

In addition, a 22 MW, 8.5 m diameter, 13.3 m EGL ball mill will be installed and will be fed by a combination of SAG mill cyclone underflow and recirculated ball mill cyclone underflow. Should the ball mill be down for relining, it will be possible to run the SAG mill in a single stage configuration, at a reduced feed rate. Dedicated liner handling machines will be provided for the SAG and ball mills to undertake mill liner changing.

17.3.6 **Gold circuit**

A gravity concentrator circuit will be included in the plant in order to recover gold. The gravity concentrate will report to rougher tables and then to cleaner tables for final upgrading. Table concentrate streams will be fed into an acid treatment vat for further upgrading. Acid treated concentrate will be washed and dried before smelting in an induction furnace to produce doré bars.

17.3.7 **Flotation**

Two banks of 5 x 300 m^3 tank cells will be used for rougher/scavenger duty. These will be configured as 1-2-2 trains with the first rougher potentially producing final grade concentrate from the high grade zones within the fast floating Kansanshi ores.

The cleaner flotation cells will be 3 x 200 m^3 tank cells. The cleaner tails will report to the cleaner/ scavenger cells. The cleaner concentrate will feed the re-cleaner cells. The re-cleaner cells will be 3 x 100 m^3 tank cells and will upgrade the cleaner concentrate and possibly the first rougher concentrate to final smelter feed grade. Re-cleaner tails will feed the cleaner cells while the final concentrate will report to the concentrate thickener via the concentrate trash screen.

The scavenger concentrate will feed the 3 x 200 m^3 cleaner/scavenger cells. The cleaner/scavenger tails will be recycled to the rougher feed or to rougher Cell 3 if required to avoid diluting the first rougher grade. The cleaner/scavenger concentrate will feed the cleaner cells.

17.3.8 **Concentrate handling**

The concentrate thickener underflow will be stored in one of two agitated stock tanks. Two sets of concentrate transfer pumps will draw from either tank and pump the concentrate approximately 4 km overland to the smelter.

Concentrate slurry will be filtered using four filters located adjacent to the smelter, two of which will be relocated from the existing plant. Filter concentrate will report either to the smelter feed conveyor or to a conveyor, feeding a storage shed. The filtrate will report to a settler to recover any concentrate contained in the filtrate or washings.

17.3.9 **Tailings disposal**

The tails from the rougher / scavenger flotation banks will gravitate to two 50 m diameter tails thickeners for water recovery. The two thickeners will be located adjacent to one another and will share an overflow process water tank. This tank will be the main source of process water for the milling circuit.

Two duty tails lines, each with two stages of centrifugal pumps (expandable to four stages) have been allowed for tailings pumping. A third line of pumps will be provided as a common standby. The pumps will be equipped with variable speed drives. The number of pump stages will be confirmed once the final pipeline route and tailings deposition plan have been developed. The tailings lines will be routed in an earth channel for containment of spillage should line leakage or failure occurs. Isolation knife gate valves will also be installed at regular intervals on the tailings line.

An agitated tails surge tank will provide a short duration surge capacity between the tailing thickeners and the final tailings pumps. It will also provide a positive suction head on the pumps.

The tailings storage facility (TSF) is an existing dam (named TSF2) currently used by the existing Kansanshi operation. For the first year of operation of the S3 process plant, the embankment forming the south east wall of TSF2 will be raised by approximately 5 m to accommodate the expected rise rate of the wall crest inside the dam. For the first one to two years of operation, it is expected that tailings discharge into the dam will be from spigot outlets on the tailings pipelines. After this time, tailings discharge will be from cyclones, to provide coarser particles for wall construction.

The design of the tailings dam is such that water can be recovered from the tailings after placement. The water will be returned back to the process water tank in the S3 process plant.

17.4 **Smelter project**

A copper smelter with a capacity to treat 1.2 million tonnes per annum of concentrate is currently ramping up operations. The Project will produce 300,000 tpa of blister copper and 1,000,000 tpa of acid. Acid supply for the increased oxide plant throughput will be provided from this new smelter.

In addition to the acid supply, other benefits of the smelter construction include:

- removal of a concentrate export levy
- saving on transport costs, i.e. blister product rather than concentrate

The main components of the smelter are as follows:

- ISA smelt furnace
- waste heat boiler
- matte settling furnace
- oxygen plant
- acid plant
- 4 x Pierce-Smith converters
- 2 x blister furnaces
- 2 x casting wheels
- concentrate receival and storage

ITEM 18 PROJECT INFRASTRUCTURE

The following information is partially reproduced and expanded upon from the December 2012 Technical Report (Journet and Cameron, DumpSolver, 2012), with an update provided by Robert Stone (QP).

18.1 Power supply

The electrical power to the Kansanshi Operations is provided by a single incoming 330kV overhead transmission line from Luano (Zesco) with a capacity of 700 MVA. Future plans include a ring feed via Lumwana at 330 kV. At Kansanshi (Zesco) there are two 65 MVA, one 80 MVA and two 120 MVA transformers which step down to 33 kV. 33 kV is fed into the process plant by means of three feeders. Another feeder supplies 33kV to the mine and two feeders supply 33kV to the smelter. There are six spare 33kV feeders which are available for expansion purposes.

The mining feeder is also used to supply certain parts of the process plant with power. This mine feeder is used for supplying in-pit skids fitted with transformers to step down to 6.6 kV (electric shovels) and 525V (dewatering pumps and drill rigs). The trolley-assist system is supplied with 33 kV directly.

Within the smelter and processing plant exist a number of 33 kV substations that supply 3 MVA and 2.5 MVA transformers (one per motor control centre) which in turn step down to 525 V which is the main low voltage used for motor and other consumer power supplies.

Large motors are supplied with other voltages such as 11 kV or 6.6 kV. Some smaller consumers are supplied with other voltages such as 11 kV by means of overhead power lines.

There is a standby generating plant designed for 20 MW but equipped with 6 MW generating capacity which is connectable to the entire 33 kV system with the exception of the smelter feeders.

18.2 Water supply and water systems

To date, process and domestic water requirements have been met primarily by mine dewatering activities, supplemented with pumping from the Solwezi River since May 2012, and for a time, with water pumped from a borefield (obsolete since May 2012). A maximum daily limit of 48 ML is currently in place for water pumped from the Solwezi River.

The current process water demand is 91.4 ML/day. This will expand to 118 ML/day once the S3 plant is completed.

The balance of make-up water will be the decant water returned from TSF1 and TSF2. Additional collection and transfer equipment will be installed at the existing TSF2 to boost the decant capacity when S3 is installed.

In order to meet a possible process water shortfall in the dry season, development of a new borefield has commenced, involving the drilling and equipping of eight new 10” to 15” diameter bores. Evaluation of alternative supply options is also in progress, and this includes increasing the limit that can be pumped from the Solwezi River.

18.3 **Air systems**

The process plants have installed a range of air systems to suit the operational requirements. These include:

- Plant and instrument air – distributed throughout the process plants for general use and to supply to air actuated instruments respectively.
- High pressure concentrate filtration air – provided for the concentrate pressure filters.
- Flotation air - low pressure air supplied to the floatation circuits.
- Blower Air – utilised in the smelter for primary smelting and copper converting.

18.4 **Fire protection system**

Fire detection and alarms are provided for all substation buildings, including the Main 33 kV substation building. The main plant will be provided with fire detection where appropriate, for example for the SX trains. All plant areas have fire hydrants strategically located for manual fire protection in addition to hand held extinguishers.

18.5 **Plant buildings**

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

18.6 **Mine services**

A new mining fleet services area has recently been constructed. The new workshops, tyre fitting, refuelling and lube facilities will suit the expanded mining operations for the next twenty years and will replace the existing facilities which will be dismantled before the eventual merger of the Main and North West Pits.

18.7 **Roads and site access**

The Kansanshi Operations site is accessed from a sealed road running north from Solwezi town, a distance of approximately 12 km. All areas of the operation including the process plants and mine sites are accessed from this road with the exception of TSF2 which is accessed by the eastern lease bypass road.

In addition, a new road has been constructed to support concentrate transfer between Sentinel and the smelter. This road exits the mine lease area to the west of Solwezi town (to avoid in-town congestion) and runs directly into the smelter complex, with a link to the main plant access road.

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

Information on the site waste dumps is provided in Item 16.1 with an accompanying plan shown in Figure 16-1. This same plan shows the location of the tailings dams, a commentary on which is provided in Item 17.3.9.

All tailings at KMP are pumped to one of two tailings storage facilities. A part of TSF1 was designed to be an acidic storage facility (and permitted as such) but is no longer in use.

The Kansanshi ore body has significant neutralising capacity and as a result, all tailings material going to TSF2 has historically been alkaline in nature. Given the dominance of neutralising material, long term acid generation at KMP is considered to be unlikely. KMP monitors the material leaving the plant and within the tailings facilities on a daily basis.

Waste rock at KMP has also been dominated by significant neutralising capacity. Domestic waste is managed according to its ability to be reused or recycled. A domestic landfill on site is managed by KMP.

There are surface tailings pipelines to be installed as part of the the S3 expansion, in addition to current pipelines. There are also two concentrate pipelines transferring concentrate to the filter plant at the smelter.

ITEM 19 MARKET STUDIES AND CONTRACTS

19.1 Markets for Kansanshi product

With the advent of on-site smelting at KMP, the supply and sale of concentrate to local smelters will end in 2015. In the future, anode product from the smelter and cathode product from the SXEW plant will be the main saleable products.

Currently, all anode and cathode product is being sold through the company's internal marketing division, Metal Corp Trading AG (MCT).

19.2 Contracts

In terms of anode product sales going forward, MCT has initiated a sales strategy focussing on a diverse and geographically spread user base in Europe, Korea, India and China. Although sales terms are under negotiation, buyers are currently contracted for approximately 70% of the expected off-take.

The anode sales costs adopted by QP Michael Lawlor for the optimisation sensitivity analyses (Item 15) and economic analysis (Item 22), were provided by MCT. Considering the relative impact that metal sales cost variances make to the marginal cut-off grade and optimal pit shell selection, this MCT information is accepted as suitable information for optimisation input and for the cashflow model which supports the Mineral Reserve estimate.

ITEM 20 ENVIRONMENTAL STUDIES, PERMITTING, SOCIAL AND COMMUNITY IMPACT

The following information has been provided by Andrew Hester, FQM Manager Environmental, Africa.

20.1 Environmental setting

The expanded S1 processing facility is located immediately adjacent to the original facility. The new smelter site is located to the immediate south west of the North West Pit. Both of these locations are in areas within the Surface Rights Boundary area, previously affected by mining and processing activities.

The location of the new S3 sulphide plant, immediately north of the existing tailings dams, has been selected based on availability of suitable space, proximity to existing infrastructure, eg existing roads, is outside the future mining envelope, and takes consideration of geotechnical conditions. The site is already disturbed from previous construction activities, apart from degraded miombo woodland in a few places.

The remaining vegetation at the site is typically open Miombo woodland vegetation, although extensively modified by previous activities. A few tree species were identified in the original Environmental Impact Statement (EIS) (Blandford, 2002) and these included the following: Syzygium guiniense afromontanum (musafwa), Parinari spp. (mupundu), Anisophyllea boehmii (mufungo), Isoberlinia angolensis (mutobo), among others. The most predominant tree species at the site were the Brachystegia species. No endangered tree species were identified.

20.2 Status of environmental approvals

The Mines and Minerals Development Act No. 7 of 2008 and its subsidiary legislation (The Mines and Minerals (Environmental) Regulations, 1997), together with the Environmental Management Act No. 12 of 2011 and its subsidiary legislation (The Environmental Protection (Environmental Impact Assessments) Regulations, 1997) each have requirements for all new projects or expansions of projects to conduct an environmental impact assessment (EIA) and submit an 'Environmental Project Brief' to the Zambia Environmental Management Agency (ZEMA) for approval before a project commences.

More than fifteen EIAs for operational infrastructure at KMP have been submitted and approved by ZEMA in the last ten years. The larger projects include:

- TSF2 – the design and construction of a new sulphide plant tailings dam facility
- smelter – the design and construction of an on-site smelter
- oxide expansion circuit – the design and construction of the oxide circuit expansion
- sulphide expansion circuit (S3) – the design and construction of the sulphide circuit expansion

20.3 **Environmental management**

The environmental and social impacts have been assessed and appropriate mitigation measures have been implemented. The EIAs 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.

Each approval is accompanied by a list of commitments. The commitments vary depending on the project, but typically require implementation of a number of control measures, adherence to related Zambian legislation and compkiance with statutory Zambian effluent and emissions limits. All commitments are captured and monitored in a site environmental register. KMP currently tracks several hundred commitments associated with the more recent EIAs.

20.4 **Resettlement**

Some of the recent developments at KMP have extended the site boundaries and local residents have been resettled involuntarily. In these cases, KMP has prepared a comprehensive Resettlement Action Plan (RAP) to guide all compensation activities. Zambia does not have any legislated guidelines on involuntary resettlement and FQM therefore ensures that all resettlement planning complies with the IFC's Performance Standards. Despite the lack of national guidelines, the RAP is still submitted to ZEMA for their consideration and approval. All KMP RAPs have been approved by ZEMA.

20.5 **Community engagement**

KMP maintains an open and respectful relationship with all local, regional and national stakeholders. Local communication includes group and one-on-one meetings with local government, traditional leaders, village elders, community elected representatives, civil society, non-governmental agencies and community members. Minutes of all meetings are recorded. Local communities have the opportunity to submit grievances to KMP via a number of channels. Grievances are investigated and resolved within a time period acceptable to each party.

20.6 **Mine closure**

The main environmental liabilities at Kansanshi 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 Kansanshi Mine closure plan is reviewed annually.

At the 31st December 2014, the unplanned asset retirement obligation (ARO) at Kansanshi was estimated to be $90.664 million. In accordance with National Legislation, Kansanshi contributes to an Environmental Protection Fund administered by the Zambian Mines Safety Department.

ITEM 21 CAPITAL AND OPERATING COSTS

21.1 Capital costs

Table 21-2 lists the capital cost provisions in the current KMP five-year forecast, as included in the Project cashflow model (item 22). These costs have been built up in detail by operations staff and counterparts in the Projects team. Several items were deleted to avoid double counting and spent costs; these were development costs (mining) and smelter ramp-up costs, respectively. Furthermore, the timing of the phase 2 smelter project was stretched out to 2020, and the timing of the S3 expansion was stretched out to 2019. These adjustments reflect the current LOM production schedule processing profile.

Table 21-1 Capital costs, 2015 to 2020

Capital Cost Items ($M)	2015	2016	2017	2018	2019	2020	Total
Operations							
Mining	34.55	29.81	12.29	8.78	-	-	85.43
Process	6.40	11.40	4.20	10.15	0.57	-	32.72
Engineering	2.98	22.97	2.80	0.10	0.43	-	29.27
Resource optimisation	0.72	0.10	0.10	0.10	0.10	-	1.12
Security	0.25	-	0.15	-	-	-	0.40
Safety	0.21	0.08	0.08	0.08	0.08	-	0.51
Commercial	0.28	0.25	0.25	0.25	0.25	-	1.28
Enviroment	0.57	0.25	0.25	0.25	0.25	-	1.57
IT	0.02	0.20	0.20	0.20	0.20	-	0.82
PR	-	0.01	0.01	0.01	0.01	-	0.04
Construction	3.06	-	-	-	-	-	3.06
Finance	0.25	-	-	-	-	-	0.25
Exploration	-	-	-	-	-	-	-
FQMO Roads projects	8.35	12.20	0.20	0.50	-	-	21.25
Subtotal	**57.63**	**77.27**	**20.52**	**20.41**	**1.88**	**-**	**177.71**
Projects							
Kansanshi smelter	(0.05)	-	-	-	-	-	(0.05)
Kansanshi smelter phase 2	85.80	100.62	134.16	167.70	134.16	48.36	670.80
Kansanshi smelter phase2 ISACONVERT	12.92	32.08	-	-	-	-	45.00
S3 expansion project	15.73	100.00	230.00	80.45	26.82	-	453.00
Power lines project	42.84	-	-	-	-	-	42.84
New plant workshop	3.59	-	-	-	-	-	3.59
Mine workshop extension	0.84	-	-	-	-	-	0.84
Other Kansanshi	11.90	15.00	-	-	-	-	26.90
Subtotal	**173.59**	**247.70**	**364.16**	**248.15**	**160.98**	**48.36**	**1,242.93**
Other							
Developtment Costs (Mining)	-	-	-	-	-	-	**-**
Smelter Plant - Ramp Up Costs	-	-	-	-	-	-	**-**
Total	**231.22**	**324.96**	**384.69**	**268.56**	**162.86**	**48.36**	**1,420.64**

21.2 Operating costs

Table 21-2 lists the total operating costs adopted for this Technical Report alongside those used in the 2012 Technical Report (Journet and Cameron, DumpSolver, 2012). The following commentary describes the derivation of these current cost inputs.

Table 21-2 Total operating costs

Operating Costs	Units	2012 NI 43-101 Report				This Technical Report		
		Oxide	Mixed	Sulphide		Oxide	Mixed	Sulphide
Variable Costs								
Ore mining differential (FQMO)	$/t process	0.47	0.47	0.47		0.57	0.57	0.57
Grade control assay costs	$/t process	0.50	0.50	0.50				
Crusher Feed Cost (FQMO)	$/t process	0.20	0.20	0.20	)			
Plant - acid (calculated in model)	$/t process				)			
Plant - direct consumables	$/t process	3.17	2.38	1.04	)>	7.88	6.48	4.32
Plant - power	$/t process	1.17	1.23	1.23	)			
Plant - other	$/t process	0.19	0.19	0.19	)			
Sub-total variable	**$/t process**	**5.70**	**4.96**	**3.63**		**8.44**	**7.05**	**4.89**
Fixed Costs								
Fixed Mining	$/t process	0.24	0.24	0.24		0.36	0.36	0.36
Engineering	$/t process	3.18	2.62	2.62		1.89	1.89	1.89
Services	$/t process	1.15	1.45	0.65		0.00	0.00	0.00
Administrative	$/t process	0.74	0.74	0.74		1.19	1.19	1.19
Labour	$/t process	1.59	0.67	0.30		1.65	0.32	0.55
Sub-total fixed	$/t process	6.90	5.72	4.55		5.10	3.77	4.00
Total Operating Costs	**$/t process**	**12.60**	**10.68**	**8.18**		**13.54**	**10.82**	**8.88**

21.2.1 Ore mining differential costs

For the pit optimisation work completed in 2014, ore mining differential (grade control) cost inputs were based on a detailed review of actual costs incurred in 2012. The resulting unit costs are listed in Table 21-3.

Table 21-3 Ore mining differential costs

Differential Ore Mining				
2012 Actuals		**Total Ore Processed**	**kt**	**24,274**
Cost Codes		**Variable Costs**		
20	280	Grade Control Lab		
		Labour	$'000	$11,594
		Stores Issues	$'000	$54
		Consumables	$'000	$108
		Other elements	$'000	$860
		O/Head Recovery	$'000	$95
		Other	$'000	$29
		Subtotal	**$'000**	**$12,740**
		Unit Cost	**$/t**	**$0.52**
2012 Actuals		**Total Ore Processed**	**kt**	**24,274**
Cost Codes		**Variable Costs**		
59	280	Grade Control		
		Stores Issues	$'000	$7
		Consumables	$'000	$922
		Other elements	$'000	$72
		Subtotal	**$'000**	**$1,001**
		Unit Cost	**$/t**	**$0.04**

21.2.2 Processing and G&A costs

For the pit optimisation work completed in 2014, process and G&A operating cost inputs were also based on the detailed review of actual costs incurred in 2012. Table 21-4 lists the processing unit costs determined for each processing feed route from the actual cost review and the adjustments that were made to correct misallocations in the accounting system and restore the relativity of the mixed and sulphide unit costs to their correct proportions.

Table 21-4 Processing costs

Common Processing				
2012 Actuals		**Total Ore Processed**	**kt**	**24,274**
Cost Codes		**Variable Costs**		
54	490	Unspecified	$'000	$86
59	440	Neutralisation	$'000	$4,532
59	450	Tails	$'000	$3,207
59	500	Processing	$'000	$2,906
		Subtotal	**$'000**	**$10,731**
		Unit Cost	**$/t**	**$0.44**
Oxide Processing				
2012 Actuals		**Oxide Ore Processed**	**kt**	**6,349**
Cost Codes		**Variable Costs**		
40	340	Crushing	$'000	$4,851
40	350	Milling	$'000	$14,460
40	360	Pre Leach Flotation	$'000	$1,485
40	370	Leach	$'000	$3,268
40	380	Post Leach Flotation	$'000	$586
40	385	Post CCD	$'000	$10,951
40	390	Flotation	$'000	$6,726
40	400	Extraction	$'000	$169
40	440	Neutralise	$'000	$569
50	513	Plant #5	$'000	$4,144
		Subtotal	**$'000**	**$47,209**
	Unit cost (incl. common processing)		**$/t**	**$7.88**
Mixed Processing				
2012 Actuals		**Mixed Ore Processed**	**kt**	**8,275**
Cost Codes		**Variable Costs**		
43	340	Crushing	$'000	$2,464
43	350	Milling	$'000	$7,230
43	390	Flotation	$'000	$11,290
		Subtotal	**$'000**	**$20,984**
	Unit cost (incl. common processing)		**$/t**	**$2.98**
		Adjusted unit cost	**$/t**	**$6.48**
Sulphide Processing				
2012 Actuals		**Sulphide Ore Processed**	**kt**	**9,650**
Cost Codes		**Variable Costs**		
41	340	Crushing	$'000	$10,268
41	350	Milling	$'000	$29,987
41	390	Flotation	$'000	$8,129
41	410	Tank farm	$'000	$781
41	440	Neutralise	$'000	$1,107
41	460	Concentrate Handling	$'000	$3,324
		Subtotal	**$'000**	**$53,596**
		Unit Cost	**$/t**	**$5.55**
		Adjusted unit cost	**$/t**	**$4.32**

Similarly, Table 21-5 lists the general and administration (G&A) fixed costs converted to equivalent variable costs, based on the actual cost review. Again, these costs have been adjusted, in this instance on the basis of future higher processing rates.

Table 21-5 G&A (administration) costs

Administration				
2012 Actuals		**Total Ore Processed**	**kt**	**24,274**
Cost Codes		**Fixed Costs**		
80	550	Site Administration	$'000	$34,282
59	540	Community	$'000	$19,948
59	560	Environmental Management	$'000	$4,037
59	570	Rehabilitation	$'000	$370
		Subtotal	**$'000**	**$58,637**
		Unit Cost	**$/t**	**$2.42**
		Adjusted unit cost	**$/t**	**$1.19**
KMP Mining				
2012 Actuals		**Total Ore Processed**	**kt**	**24,274**
Cost Codes		**Fixed Costs**		
59	260	Technical Services	$'000	$2,034
20	270	Surveying	$'000	$478
59	280	Grade Control	$'000	$2,193
20	290	Geo Monit&Plan	$'000	$10,274
20	330	Dewatering	$'000	$1,638
		Subtotal	**$'000**	**$16,617**
		Unit Cost	**$/t**	**$0.68**
		Adjusted unit cost	**$/t**	**$0.36**
Engineering				
2012 Actuals		**Total Ore Processed**	**kt**	**24,274**
Cost Codes		**Fixed Costs**		
59	305	Roads	$'000	$93
59	519	Water Supply	$'000	$2,569
59	520	Workshops	$'000	$12,560
59	530	Stores	$'000	$20,385
59	545	Services	$'000	$39,902
		Subtotal	**$'000**	**$75,509**
		Unit Cost	**$/t**	**$3.11**
		Adjusted unit cost	**$/t**	**$1.89**

21.2.3 **Labour costs**

Table 21-6 lists the labour costs converted to equivalent variable costs, based on the actual cost review. These costs have also been adjusted, in this instance again on the basis of future higher processing rates.

Table 21-6 Labour costs

Common Processing				
2012 Actuals		**Total Ore Processed**	**kt**	**24,274**
Cost Codes		**Labour Costs**		
54	490	Gold Plant - Unspecified		$1,064
59	440	Neutralisation		$772
59	450	Tails		$11,450
59	500	Processing		$3,382
		Subtotal	**$'000**	**$16,668**
		Unit Cost	**$/t**	**$0.69**
		Adjusted unit cost	**$/t**	**$0.32**
Oxide Processing				
2012 Actuals		**Total Ore Processed**	**kt**	**6,349**
Cost Codes		**Labour Costs**		
40	340	Crushing		$4,384
40	350	Milling		$3,343
40	360	Pre Leach Flotation		$268
40	370	Leach		$174
40	380	Post Leach Flotation		$40
40	385	Post CCD		$1,877
40	390	Flotation		$1,539
40	400	Extraction		$652
40	420	SX		$257
40	430	EW		$5,148
40	440	Neutralise		$29
50	513	Acid Plant #5		$160
		Subtotal	**$'000**	**$17,871**
		Unit Cost	**$/t**	**$2.81**
		Adjusted unit cost	**$/t**	**$1.33**
Mixed Processing				
2012 Actuals		**Mixed Ore Processed**	**kt**	**8,275**
Cost Codes		**Labour Costs**		
43	340	Crushing		$0
43	350	Milling		$22
43	390	Flotation		$8
		Subtotal	**$'000**	**$30**
		Unit Cost	**$/t**	**$0.00**
		Adjusted unit cost	**$/t**	**$0.00**
Sulphide Processing				
2012 Actuals		**Sulphide Ore Processed**	**kt**	**9,650**
Cost Codes		**Labour Costs**		
41	340	Crushing		$1,822
41	350	Milling		$2,129
41	390	Flotation		$200
41	410	Tank farm		$69
41	440	Neutralise		$5
41	460	Concentrate Handling		$351
		Subtotal	**$'000**	**$4,576**
		Unit Cost	**$/t**	**$0.47**
		Adjusted unit cost	**$/t**	**$0.23**

21.2.4 **Mining costs**

A set of linear mining cost algorithms were developed for the 2012 Technical Report (Journet and Cameron, DumpSolver, 2012) to provide incremented ore and waste mining costs by depth for the purposes of pit optimisation input.

These algorithms were updated for the 2014 optimisation based on indicative ore and waste haul profiles for the Main, North West and South East Dome pits within the then current five year production plan, and with reference to current mining costs provided by FQMO (Table 21-7). These FQMO cost inclusions take account of:

- fuel consumption for drilling rigs, shovels/excavators and trucks
- tyre consumption for trucks
- explosives and mining equipment maintenance consumables
- operator wages
- maintenance and FQMO administration fixed costs
- haulage to initial concept IPC locations in Main Pit
- trolley-assist waste haulage along routes within the Main Pit five year production plan

Table 21-7 Mining costs

Item		Unit	Cost
Fixed costs			
	Drill and blast	$	1,510,000
	Load and haul	$	6,268,378
	Other	$	9,800,000
Variable costs			
	Drill and blast	$/bcm	1.88
	Load and haul	$/bcm	0.28

From this information, the variable relationships for mining costs as used for pit optimisation were as follows:

- ore mining ($/t) = -0.0019 x RL + 3.95
- waste mining ($/t) = -0.0047 x RL + 7.51

The weighted average variable mining costs are $3.00/t for ore and $3.13/t for waste.

21.2.5 **Metal costs**

In addition to royalties, metal costs for each of the copper and gold products comprise concentrate transport charges, deductions (1 g/t gold from concentrate), smelter treatment charges and refining charges. For the pit optimisation work completed in 2014, metal cost inputs were also based on the detailed review of actual costs incurred in 2012, and assumed the sale of copper-gold concentrates (plus cathode) and refining off-site. Table 21-8 lists the revised costs, inclusive of metal costs adjusted for smelting charges and the sale of anode. The smelter and anode adjustment details are:

- smelter recovery = 95.7%
- Cu grade in anode = 99.5%

- Au grade in anode = 10g/t
- refining deductions = 0.3% Cu and 1 g/t Au
- percentage payable = 199.2%
- smelting cost = $75/dmt
- anode charge (penalty) = $140/t
- anode freight charge = $301.50/t

21.2.6 **Other costs**

The optimisations did not include a stockpile reclaim cost in the total processing costs. For completeness, a figure of $1.30/t is estimated for inclusion in the cashflow model of Item 22 (ie, an additional cost that equates to about 2% of the total operating costs).

Sustaining capital costs were not considered for the optimisations, and so an allowance of 5% of the total operating costs has been included in the cashflow model.

A closure cost provision of $90.7M (Item 20.6) is also included in the cashflow model, and this has been spread notionally over the three final years of the Project life.

Table 21-8 Metal costs

Copper Metal Costs	Units	2014 Optimisation			2015 Cashflow Model		
		Leach	Mixed	Sulphide	Leach	Mixed	Sulphide
Cathode Metal Costs							
SX+EW consumables	$/t cathode	214.84			214.84		
EW power	$/t cathode	175.40			175.40		
Realisation/freight	$/t cathode	0.00			0.00		
Royalties	$/t cathode	396.81			595.25		
Total Cathode Cu Metal Costs	$/t cathode	787.06			985.49		
Total Cathode Cu Metal Costs	**$/t Cu in cathode**	**0.36**			**0.45**		
Concentrate Metal Costs							
Concentrate Grade	%	26.5%	26.5%	26.5%			
Moisture Content	%	10.0%	10.0%	10.0%			
Kansanshi TCRCs:							
Concentrate transport (wet)	$/t conc	0.00	0.00	0.00			
Treatment cost (wet)	$/t conc	63.50	63.50	63.50			
Refining charge	c/lb	6.35	6.35	6.35			
Realisation/freight	$/t Cu in conc.	0.00	0.00	0.00			
Sub total	$/t Cu in conc.	406.24	406.24	406.24			
Royalties (Gross)	$/t Cu in conc.	396.81	396.81	396.81			
Total Concentrate Cu Metal Cost	$/t Cu in conc.	803.05	803.05	803.05			
Total Concentrate Cu Metal Cost	**$/lb Cu**	**0.36**	**0.36**	**0.36**			
Anode Metal Costs							
Concentrate Grade	%				26.5%	26.5%	26.5%
Moisture Content	%				10.0%	10.0%	10.0%
Kansanshi TCRCs:							
Copper payable in anode	%				99.2%	99.2%	99.2%
Smelting cost (wet)	$/t conc				67.50	67.50	67.50
Refining charge	c/lb				0.90	0.90	0.90
Transport charge	$/t anode				301.50	301.50	301.50
Anode discount	$/t anode				140.00	140.00	140.00
Sub total (payable)	$/t Cu in anode				715.76	715.76	715.76
Royalties (Gross)	$/t Cu in anode				595.25	595.25	595.25
Total Anode Cu Metal Cost	$/t Cu in anode				1,311.01	1,311.01	1,311.01
Total Anode Cu Metal Cost	**$/lb Cu**				**0.59**	**0.59**	**0.59**
Weighted average float/leach Cu cost	**$/lb Cu**	**0.36**	**0.36**	**0.36**	**0.54**	**0.59**	**0.59**
Gold Metal Costs (Con/Anode)	**Units**	**2014 Optimisation**			**2015 Cashflow Model**		
		Leach	**Mixed**	**Sulphide**	**Leach**	**Mixed**	**Sulphide**
Concentrate Metal Costs							
Average copper recovery	%		74.0%	89.0%			
Concentrate pull factor	%		2.01%	2.05%			
Au grade in concentrate	g/t Au		2.48	2.93			
Smelter deduction	g/t Au		1.00	1.00			
Smelter deduction value	$/oz	233.00	484.34	409.03			
Royalties (Gross)	$/oz	72.00	72.00	72.00			
Total Concentrate Au Metal Cost	**$/oz**	**305.00**	**556.34**	**481.03**			
Anode Metal Costs							
Au grade in concentrate	g/t Au				10.00	10.00	10.00
Smelter deduction	g/t Au				1.16	1.16	1.16
Smelter deduction value	$/oz				139.20	139.20	139.20
Royalties (Gross)	$/oz				108.00	108.00	108.00
Total Anode Au Metal Cost	**$/oz**				**247.20**	**247.20**	**247.20**
Gold Metal Costs (Dore)	**Units**	**2014 Optimisation**			**2015 Cashflow Model**		
		Leach	**Mixed**	**Sulphide**	**Leach**	**Mixed**	**Sulphide**
Refining Cost	$/oz Au	20.00	20.00	20.00	20.00	20.00	20.00
Royalties (Gross)	$/oz Au	72.00	72.00	72.00	108.00	108.00	108.00
Total Gravity Au Metal Cost	**$/oz Au**	**92.00**	**92.00**	**92.00**	**128.00**	**128.00**	**128.00**

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 is intended to support the Mineral Reserve estimate, and in order to demonstrate a positive cashflow for each year of mining and processing. The development capital costs and longer term closure costs are included in the analysis for completeness.

The 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 model presented in Table 22-1.

22.2 Production schedule

The production schedule forming the basis of the cashflow model is the same as that listed in Table Table 16-3 of Item 16.

22.3 Cashflow model

The cashflow model to support the Mineral Reserve estimate is listed in Table 22-1. The annual revenues are calculated from the same metal prices as used in the pit optimisation process (Item 15):

- Copper = US$3.00/lb (US$6,615/t)
- Gold = US$1,200/oz

The capital costs are the same as those listed in Item 21 and represent full year expenditure in 2015, as against partial year revenue and operating expenditure fir the same year.

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

- Mining waste = $3.13/t waste (overall average for all pits)
- Mining ore = $3.00/t ore (overall average for all pits)
- Processing costs = $13.54/t for mixed leach ore, $10.82/t for mixed float ore and $8.88/t for sulphide ore

An additional cost of $1.30/t ore reclaimed has been adopted for reclaim from longer term ore stockpiles.

The metal costs (including TCRC's and royalties) equate to the same costs as summarised in Item 21, now adjusted to reflect the sale of cathode and anode products:

- total cathode copper metal costs = $0.45/lb Cu

- total copper in anode metal costs = $0.59/lb Cu
- average float/leach feed metal costs = $0.54/lb Cu
- total gold in copper concentrate metal costs = $247.20/oz float/leach feed and for sulphide float feed (based on overall average head grades and copper recovery values)
- total gold in gravity circuit metal costs = $128.00/oz

The recovery values shown in Table 22-1 are average figures resulting from the application, on a mining model block by block basis, of the variable relationships listed in Item 15.

22.4 **Sensitivity analysis**

A sensitivity analysis was completed as part of the pit optimisation work described in Item 15.4.5. The most sensitive variable is metal price (and recovery, since the magnitude of impact is the same). Based on the undiscounted cashflow model, Table 22—2 summarises the impact of varying the copper price, mining costs, operating costs and metal costs by +/- 10% and confirms this analysis.

The +/-10% metal cost variance shown in Table 22-2 reflects the impact of TCRC variances, and also the implicit variance of the royalty rate between 7.1% and 11.9%.

Table 22-1 Mineral Reserves cashflow model summary

	UNITS		TOTAL	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
MINING																				
Total ore	Tonnes	Mt	**668.03**	30.75	60.19	41.47	63.33	56.88	51.93	53.34	58.54	61.62	36.91	33.87	45.81	34.65	23.00	10.74	4.07	0.93
Total waste	Tonnes	Mt	**1,451.18**	82.11	97.68	130.05	130.63	120.38	143.01	141.60	128.28	133.32	93.53	69.04	84.92	50.76	29.42	13.02	3.06	0.40
Strip ratio			**2.17**	2.67	1.62	3.14	2.06	2.12	2.75	2.65	2.19	2.16	2.53	2.04	1.85	1.47	1.28	1.21	0.75	0.43
Total reclaim	**Tonnes**	**t**	**232.38**	**0.00**	**2.13**	**5.98**	**7.26**	**5.36**	**7.45**	**8.42**	**4.60**	**11.32**	**27.70**	**22.18**	**15.23**	**20.23**	**32.79**	**37.11**	**17.01**	**7.60**
TOTAL FEED TO PLANT (after mining dil'n & recovery)																				
Mixed leach/float	Tonnes	Mt	**251.21**	9.88	15.56	15.56	15.56	15.56	15.56	15.56	15.56	15.56	15.56	15.56	15.56	15.56	15.56	15.56	15.56	7.92
	Cu	%	**0.89**	1.27	1.15	1.18	1.21	1.20	0.95	0.89	1.02	0.86	0.94	0.80	0.76	0.68	0.63	0.59	0.60	0.30
	Au	g/t	**0.12**	0.14	0.15	0.12	0.16	0.09	0.12	0.10	0.11	0.12	0.12	0.13	0.12	0.13	0.12	0.13	0.13	0.13
Sulphide float	Tonnes	Mt	**470.43**	5.99	13.17	19.35	25.54	25.54	38.10	38.10	38.10	38.10	38.10	38.10	38.10	38.10	38.10	32.20	5.17	0.54
	Cu	%	**0.58**	0.91	0.87	0.84	0.84	0.83	0.69	0.69	0.69	0.65	0.53	0.41	0.45	0.45	0.35	0.27	0.38	0.59
	Au	g/t	**0.12**	0.09	0.13	0.13	0.16	0.13	0.12	0.12	0.12	0.10	0.11	0.10	0.09	0.11	0.11	0.12	0.12	0.24
Total feed	**Tonnes**	**Mt**	**721.63**	**15.86**	**28.73**	**34.91**	**41.10**	**41.10**	**53.67**	**53.67**	**53.67**	**53.67**	**53.67**	**53.67**	**53.67**	**53.67**	**53.67**	**47.77**	**20.73**	**8.45**
	Cu	**%**	**0.69**	**1.14**	**1.02**	**0.99**	**0.98**	**0.97**	**0.76**	**0.75**	**0.78**	**0.71**	**0.65**	**0.52**	**0.54**	**0.52**	**0.43**	**0.38**	**0.55**	**0.32**
	Au	**g/t**	**0.12**	**0.12**	**0.14**	**0.13**	**0.16**	**0.11**	**0.12**	**0.11**	**0.12**	**0.10**	**0.11**	**0.11**	**0.10**	**0.12**	**0.11**	**0.13**	**0.13**	**0.14**
AVERAGE RECOVERIES																				
Mixed float/leach	Cu	%	**77.9%**	79.8%	79.3%	79.0%	78.0%	79.5%	77.4%	77.3%	78.4%	78.4%	78.0%	76.9%	77.5%	79.5%	62.7%	79.8%	79.7%	79.7%
	Au	%	**66.2%**	65.3%	65.6%	65.5%	65.9%	65.9%	66.3%	66.3%	66.2%	66.4%	66.4%	66.5%	66.5%	66.3%	66.6%	66.4%	66.4%	67.4%
Sulphide float	Cu	%	**89.4%**	87.9%	88.5%	89.4%	86.6%	88.6%	90.1%	89.9%	89.8%	89.8%	88.9%	89.2%	90.1%	90.5%	90.0%	89.8%	89.5%	88.4%
	Au	%	**63.4%**	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%	63.4%
Overal average	Cu	%	**84.1%**	**82.4%**	**83.3%**	**83.8%**	**82.4%**	**84.1%**	**85.0%**	**85.5%**	**85.9%**	**85.5%**	**84.3%**	**83.4%**	**84.8%**	**86.8%**	**78.2%**	**84.8%**	**81.3%**	**80.8%**
	Au	%	**64.4%**	**64.8%**	**64.6%**	**64.3%**	**64.4%**	**64.2%**	**64.3%**	**64.1%**	**64.2%**	**64.4%**	**64.4%**	**64.5%**	**64.5%**	**64.4%**	**64.4%**	**64.4%**	**65.7%**	**67.0%**
METAL RECOVERED (after mining dil'n & recovery)																				
Mixed float/leach	Cu	kt	**1,749.8**	100.4	142.4	144.3	147.6	149.0	114.2	107.7	124.7	104.9	114.3	95.9	91.5	84.4	61.7	73.3	74.3	19.2
	Au	koz	**666.5**	28.7	47.9	40.0	51.5	29.9	40.5	32.5	37.4	39.0	41.1	43.1	40.0	43.5	41.2	44.1	43.1	23.0
Sulphide float	Cu	kt	**2,443.9**	48.3	101.5	145.9	184.2	187.3	234.6	236.1	236.7	221.7	178.8	138.9	154.6	156.5	119.6	79.0	17.6	2.8
	Au	koz	**1,105.7**	11.5	35.1	52.0	81.2	66.7	93.8	93.5	92.2	75.8	86.2	79.0	72.7	86.9	84.3	79.9	12.2	2.7
Total recovered	Cu	kt	**4,193.7**	148.8	243.8	290.2	331.8	336.3	348.8	343.7	361.4	326.6	293.0	234.8	246.0	240.9	181.3	152.3	91.9	22.1
	Au	koz	**1,772.2**	40.2	83.1	92.0	132.7	96.5	134.3	126.0	129.6	114.8	127.3	122.1	112.6	130.4	125.5	124.0	55.3	25.7
GROSS REVENUE																				
Cu $/t	6,613.86	$M	**27,736.4**	983.8	1,612.8	1,919.2	2,194.5	2,224.3	2,307.1	2,273.3	2,390.3	2,159.8	1,938.2	1,552.8	1,627.2	1,593.3	1,199.0	1,007.1	607.7	145.9
Au $/oz	1,200.00	$M	**2,126.7**	48.2	99.7	110.4	159.3	115.8	161.1	151.2	155.5	137.8	152.8	146.5	135.2	156.5	150.6	148.8	66.4	30.8
subtotal		**$M**	**29,863.1**	**1,032.1**	**1,712.5**	**2,029.6**	**2,353.8**	**2,340.2**	**2,468.2**	**2,424.6**	**2,545.8**	**2,297.6**	**2,090.9**	**1,699.3**	**1,762.4**	**1,749.7**	**1,349.6**	**1,155.9**	**674.1**	**176.7**
CAPITAL COSTS																				
Operations capital		$M	**177.7**	57.6	77.3	20.5	20.4	1.9	0.0											
Projects capital		$M	**1,242.9**	173.6	247.7	364.2	248.2	161.0	48.4											
Sustaining capex (5% of total opex)		$M	**703.9**	26.5	40.2	45.5	51.4	48.7	57.0	57.0	55.5	57.1	48.2	43.5	47.4	40.8	36.3	29.7	13.8	5.3
Closure and reclamation		$M	**90.7**															30.2	30.2	30.2
subtotal		**$M**	**2,215.2**	**257.7**	**365.1**	**430.1**	**320.0**	**211.6**	**105.3**	**57.0**	**55.5**	**57.1**	**48.2**	**43.5**	**47.4**	**40.8**	**36.3**	**59.9**	**44.0**	**35.6**
OPERATING COSTS																				
Mining																				
Ore $/t mined	3.00	$M	**2,004.1**	92.2	180.6	124.4	190.0	170.7	155.8	160.0	175.6	184.9	110.7	101.6	137.4	103.9	69.0	32.2	12.2	2.8
Waste $/mined	3.13	$M	**4,542.2**	257.0	305.7	407.1	408.9	376.8	447.6	443.2	401.5	417.3	292.8	216.1	265.8	158.9	92.1	40.7	9.6	1.2
Processing (incl G&A)																				
Mixed-leach $/t process	13.54	$M	**1,573.4**	100.0	141.5	143.0	116.5	121.0	90.7	91.0	97.1	83.4	82.9	79.8	81.7	90.0	74.2	84.3	86.2	10.2
Mixed-float $/t process	10.82	$M	**1,467.5**	27.3	55.8	54.5	75.7	72.1	96.3	96.1	91.2	102.1	102.6	105.0	103.5	96.9	109.5	101.4	99.9	77.7
Sulphide-float $/t process	8.88	$M	**4,188.6**	53.3	117.2	172.3	227.4	227.4	339.3	339.3	339.3	339.3	339.3	339.3	339.3	339.3	339.3	286.7	46.0	4.8
Stockpile reclaim $/t ore	1.30	$M	**302.1**	0.0	2.8	7.8	9.4	7.0	9.7	10.9	6.0	14.7	36.0	28.8	19.8	26.3	42.6	48.2	22.1	9.9
subtotal		**$M**	**14,078.0**	**529.7**	**803.5**	**909.1**	**1,027.9**	**974.9**	**1,139.4**	**1,140.5**	**1,110.7**	**1,141.7**	**964.2**	**870.6**	**947.5**	**815.3**	**726.6**	**593.7**	**276.0**	**106.6**
METAL COSTS (INCLUDING 9% ROYALTIES)																				
Mixed float/leach																				
Total Cu Metal Cost		$M	**2,066.1**	118.6	168.1	170.4	174.3	175.9	134.9	127.1	147.3	123.8	135.0	113.2	108.0	99.7	72.8	86.6	87.7	22.7
Total Au Metal Cost		$M	**332.5**	18.0	26.2	25.7	27.5	3.5	21.5	19.5	22.5	19.8	21.4	19.0	18.0	17.4	13.8	15.8	15.8	5.5
subtotal		**$M**	**2,398.7**	**136.5**	**194.4**	**196.1**	**201.8**	**179.4**	**156.4**	**146.6**	**169.8**	**143.6**	**156.4**	**132.2**	**126.0**	**117.1**	**86.6**	**102.4**	**103.6**	**28.2**
subtotal (excluding royalties)		**$M**	**1,149.5**	**67.1**	**95.2**	**96.5**	**98.7**	**77.8**	**76.5**	**72.0**	**83.4**	**70.2**	**76.5**	**64.2**	**61.3**	**56.6**	**41.4**	**49.2**	**49.9**	**13.0**
Sulphide float																				
Total Cu Metal Cost		$M	**3,204.0**	63.4	133.0	191.3	241.5	245.6	307.5	309.5	310.3	290.7	234.3	182.1	202.6	205.1	156.8	103.5	23.1	3.7
Total Au Metal Cost		$M	**480.5**	8.3	18.7	27.1	36.0	34.9	44.9	44.9	44.6	41.0	35.8	29.2	30.8	32.3	26.9	20.4	4.0	0.7
subtotal		**$M**	**3,684.5**	**71.6**	**151.7**	**218.3**	**277.5**	**280.5**	**352.4**	**354.4**	**354.8**	**331.6**	**270.1**	**211.3**	**233.4**	**237.5**	**183.8**	**124.0**	**27.0**	**4.4**
subtotal (excluding royalties)		**$M**	**2,110.3**	**41.6**	**87.5**	**125.9**	**159.1**	**161.8**	**202.6**	**203.8**	**204.0**	**191.5**	**154.4**	**120.1**	**133.5**	**135.0**	**103.5**	**68.3**	**15.2**	**2.4**
CASHFLOW		**$M**	**7,486.7**	**36.4**	**197.8**	**275.9**	**526.6**	**693.8**	**714.7**	**726.0**	**854.9**	**623.6**	**652.0**	**441.6**	**408.2**	**539.2**	**316.3**	**276.0**	**223.5**	**1.9**

Table 22—2 Undiscounted cashflow model sensitivity analysis

Cu price		$M	
2.70	$/lb	4,713.1	63%
3.00	$/lb	7,486.7	100%
3.30	$/lb	10,260.4	137%
Mining costs		**$M**	
2.78	$/t mined	8,174.1	109%
3.09	$/t mined	7,486.7	100%
3.40	$/t mined	6,799.3	91%
Operating costs		**$M**	
9.39	$/t processed	8,277.5	111%
10.44	$/t processed	7,486.7	100%
11.48	$/t processed	6,695.9	89%
Cu metal costs		**$M**	
0.59	$/lb	8,095.0	108%
0.66	$/lb	7,486.7	100%
0.72	$/lb	6,878.4	92%

ITEM 23 **ADJACENT PROPERTIES**

There are no adjacent properties or relevant information pertaining to adjacent properties that are material to this Technical Report.

ITEM 24 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant information or explanation required to make this Technical Report understandable and not misleading.

ITEM 25 INTERPRETATIONS AND CONCLUSIONS

25.1 Mineral Resource modelling and estimation

The Kansanshi North West, Main and South East updated Mineral Resource estimates were supported by extensive drillhole coverage. Open pit mining has exposed the prevailing deposit geology at the North West and Main deposits, providing valuable support for the estimates under-pinning geology and mineralisation models. QAQC of samples demonstrates representative sample assay results from diamond drilling and the last two years of RC drilling. Geological understanding and resulting models are good and conform with the extensive deposit geology exposed in current pits. The large number of close spaced RC holes adds confidence to local geology and grade continuity of the respective domains of mineralisation. The Qualified Person, David Gray believes this updated Mineral Resource estimate to be representative of the prevailing geology and drilled sample data.

25.1.1 Procedures

Based on FQM drilling completed to date, the industry standard procedures applied, verification of data by FQM and CSA, the sound consideration of prevailing geology from pit exposures and relevant resource estimation methods employed, it is the QP's opinion that the Mineral Resource stated in this report comply with the reporting requirements of the Canadian National Instrument 43-101 guidelines: 'Standards of Disclosure for Mineral Properties' of April 2011.

25.1.2 Database validation

FQM have dedicated database administrators and data is hosted in an SQL database which has built-in validations and constraints. The database export provided had no significant issues with any of the included data, but no geological or metadata records (hole diameter, recoveries, etc.) were provided for the grade control holes. Preliminary validation of RC grade control data raised a number of issues regarding duplication of drill hole names and drill hole locations. This had no impact on the Mineral Resource estimate since RC grade control data was not used in the estimation of stratigraphy. No such issues were found in the comparably smaller RC grade control dataset used in the vein estimate.

25.1.3 Data

The dataset used in the Mineral Resource estimate is substantial, comprising 1,109 exploration diamond drill holes and 5,160 RC grade control holes in the vein estimate. Currently, the spacing of diamond drilling is insufficient to adequately define continuity of veins, supporting the inclusion of RC grade control holes. The remaining part of the Mineral Resource estimate only used exploration diamond drilling comprised of 3,265 drill holes.

25.1.4 Sample bias – RC vs DD in NW

Bias risks between samples using different drilling methods and field sample masses were investigated by the QP with some risk of bias identified between the RC and diamond drilled samples. Samples used in these comparisons were however not spatially twinned. As such further studies need to be completed in order to quantify the degree of this bias.

25.1.5 **Geological logging**

Geological logging of exploration and resource definition data appears to be of good quality. Material was defined as vein if LITH1 was logged as vein or if Vein_Pct > 60%. This was considered adequate to define veins. RC grade control drilling had not been logged, and therefore assumptions had to be made regarding high grade sample data that fell within a certain zone of influence around the vein wireframes.

25.1.6 **Geological Model**

The geological model has changed considerably since the 2012 Mineral Resource estimate. Risks and recommendations arising from the 2012 Technical Report involving the need to reduce the amount of dilution in veins, quantifying the continuity of veins, particularly in stratigraphic units where veins are less well developed, and restricting the horizontal influence of high grade vein samples in the stratigraphy all lead to the change in technique. Veins were modelled only in MMC and UMC, units where data broadly supports the continuity. The wireframing involved digitising a centreline through high grade vein mineralisation, and modelling a consistent width, which tapered with depth.

Veins, though present in other stratigraphic units, were understood to be less well developed and were not wireframed, due to poor data support, and lack of confidence in interpreting continuity.

Weathering surfaces delineated saprolite, saprock and fresh material. The weathering surfaces did not extend all the way to the edge of the model in the north western portion of the NW. This affects approximately 40,000 blocks, which in the context of the resource, is not material. Assumptions based on RL were made to define saprock and fresh material in this area, so that density and GAC could be assigned to all blocks.

25.1.7 **Oxidation and Material Types**

A different modelling approach was applied to oxidation materials in order to assist with metallurgical definition of materials, and to reduce the subjectivity of digitising surfaces. Refractory was digitised as previously, and one surface was digitised where ratios of AsCu/TCu were in the order of 0.1/0.2 to define the transition from oxide/transitional to predominantly primary sulphide material. Ratios were then used to define what was highly leachable, leachable, possibly leachable/possibly flotation and definitely flotation. The OXMAT field was used to define these material types and the following ratios were used:

- RATIO≥0.8 = high grade carbonate material: highly leachable;
- 0.4≤RATIO<0.8 = Oxide: leachable;
- 0.1<RATIO<0.4 = Transitional – may be leachable or better suited to flotation circuit;
- RATIO≤0.1 = Sulphide – flotation circuit.

25.1.8 **Bulk Density and Gangue Acid Consumption**

Additional data was collected since 2012 which lead to the revision of bulk density and GAC values for different units. This has resulted in an improvement in the confidence of the tonnage of the resource. Units with missing data were assigned mean values from other units, with assumptions made about their validity.

25.1.9 **Assay QA/QC**

The QA/QC programme implemented at Kansanshi for the exploration samples was rigorous and comprehensive. Sufficient QA/QC material has been included with the drill samples to give confidence that assay results obtained should accurately reflect the sample grades. Ongoing monitoring occurs with monthly QA/QC reports generated in QA/QCR; issues were investigated in a timely manner and although some issues are ongoing, no fatal flaws are apparent.

Some of the diamond assay results might not be representative of the samples as failed CRMs are re-assayed, but not the associated samples (from the same digestion batch). This could potentially mask the fact that there were issues with the QA/QC and samples as the corrected re-assayed values for the CRMs have been included in the database. However, enough checks have been undertaken to ensure that the assay results achieved should be representative of the samples used.

QA/QC checks were a relatively recent addition in the RC grade control drilling with no effective QA/QC undertaken prior to February 2013. Thus any results prior to this date would have to be viewed with caution despite been located with mined out areas.

25.1.10 **Mineral Resource Estimate**

The grade estimate was completed in three stages. Firstly, the veins were estimated into a 20m x 20m x 10m block model using Length Weighted Inverse Distance Weighting to the power of one (IDW1). Hard boundaries were used between oxidation units. Veins were localised to 5m x 5m x 5m SMUs, to honour the volume estimated by the wireframes.

The stratigraphy was estimated using Uniform Conditioning into 40m x 40m x 5m blocks and localised to 10m x 10m x 5m SMU's. Hard boundaries were used between stratigraphic units, while soft boundaries were used between oxidation units in each respective stratigraphic unit.

Finally, the proportion of a block that had vein material that had not been wireframed was estimated using Indicator Kriging. The mean grade was assigned to that part of the block that had >50% vein. The final grade of the block comprised the combination of stratigraphy and un-wireframed vein, weighted by their respective proportions.

Block grade estimates were validated, combined and classified. Portions of North West and Main were classified as Measured, Indicated and Inferred, while South East Dome was classified as Indicated and Inferred. Classification was guided by confidence in geology model and estimation methodology which informs volume, drill hole spacing, QA/QC, and confidence in the grade estimate which was informed by Search Pass and Slope. Wireframes were created for Measured (Search Pass 1 or 2 and Slope>0.7) and Indicated (Search Pass 1 or 2 and Slope > 0.5), and remaining material classified as Inferred. Diorite was classified as Inferred. Laterite was not classified, in line with the 2012 MRE.

There was a 19% drop in tonnes and a 6% increase in TCu grade from the December 2012 to the May 2015 Mineral Resources. 10% of the reduced tonnages and copper metal is due to mining since December 2012. Accordingly, changes to this resource estimate have resulted in a 9% drop in tonnes, a 6% increase in TCu grade and a 5% loss in total copper metal. This is mainly due to the improved vein and stratigraphy estimation methods together with the confidence increases and extensions from South East dome drilling

A regularised block model to 10m x 10m x 5m was provided to FQM for pit optimisation and mine planning work.

25.2 **Mineral Reserve estimation**

The Mineral Reserve estimate for the Kansanshi Operations is 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 variable processing recovery relationships determined from metallurgical testwork and analysis. Unplanned mining dilution and mining recovery were considered in the optimisation and scheduling process. Between the optimisations completed in 2014 and the cashflow modelling completed to support the Mineral Reserves estimate at May 2015, operating and metal costs have been reviewed and updated for inclusion in the cashflow model.

The pit designs take account of the desired 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 mine site layout plan and haulage simulations have been undertaken to provide incremental mining costs which reflect haulage to in-pit crusher positions and waste haulage under trolley-assist.

In the opinion of Michael Lawlor (QP), therefore, the Mineral Reserve estimate reflects an achievable mining plan and production sequence, and one which has taken account of staged mining and processing capacity, reasonable total material movements (and hence equipment usage) profiles, and longer term in-pit crusher relocations.

The commissioning of the first phase KMP smelter has reduced the tendency for concentrate stockpiling due to third-party smelting capacity constraints. Furthermore, the by-production of sulphuric acid from the smelter brings a benefit to the continued SXEW processing of mixed leach ore. Lesser quality ores with high gangue acid consumption levels can now be processed without incurring an acid procurement cost.

25.2.1 **Mining**

There is considered to be minimal risk attributable to the mining method and primary equipment in use at the Kansanshi Operations. The method and equipment items are conventional and suitable for a large scale bulk mining project.

25.2.2 **Processing**

By virtue of the adopted conventional processing technology and operational performance to date, in conjunction with the adoption of variable process recovery relationships, there is considered to be minimal risk attributable to the future processing of Kansanshi ores. Furthermore, the flexibility of the circuitry allows any ore type to be treated through any of the three circuits. Long and short term mine planning now takes cognisance of the ability to optimise plant feed and maximise net return.

25.2.3 Costs of production

The risk of significantly increased Zambian government royalties has been dealt with by scaling back expenditure on the smelter and the S3 expansion project, both of which will now be staged to ultimate capacity. Despite the obvious cashflow outcome, the imposition of a new 9% royalty rate has been demonstrated as having neglible impact on the extent to which metal costs contribute to cut-off grade parameters and the Mineral Reserve estimate.

Switching from cathode plus concentrate production to cathode plus anode production, also appears to have minimal impact on the cut-off grades and selection of the optimal pit for mine planning. On this basis, Michael Lawlor (QP) is of the opinion that the metal cost inputs used for the Mineral Reserve estimate are reasonable, with variance of these inputs having less of an impact than the operating cost inputs.

The operating costs estimates have been derived from a review of actual costs, adjusted for future scales of production. In the case of mining costs, these have been derived from first principles estimates accounting for simulated ore and waste haulage profiles. Whilst the Mineral Reserve estimates are more sensitive to metal price changes and to processing recovery, it is the opinion of Michael Lawlor (QP) that operating cost reviews should be a continuous improvement action item, in order to reduce the uncertainty and improve the accuracy of cost estimates for future mine planning.

25.2.4 Environmental compliance

Numerous environmental impact assessments for operational infrastructure at KMP have been submitted and approved by the Zambia Environmental Management Agency in the last ten years. The environmental and social impacts have been assessed and appropriate mitigation measures have been implemented. The EIAs 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.

Furthermore, the Company is implementing a number of environmental standards in accordance with the ISO 14001 Environmental Management System. To the Company's knowledge, KMP's operations are not considered by any applicable environmental regulatory authority to be a risk to the environment.

25.2.5 Permitting

KMP is the owner of the LML 16 mining licence, which was issued on the 7th March 1997 and is valid until 7th March 2022 whereupon renewal can be applied for.

The licence area was expanded in 2014 and with the added capacity for in-pit backfilling in the North West Pit, there is no longer considered to be an issue of waste dumping volumes and constraints of available space.

ITEM 26 RECOMMENDATIONS

The KMP property is in production and staged material exploration and engineering studies for the development of the Operations have been concluded. Whilst expansion projects and technical enhancements have been described in this Technical Report, the advancement of these are not contingent upon the recommendations provided under this Item. The recommendations provided herein are in the context of continuous improvements, benefiting future Mineral Resource and Mineral Reserve estimates.

26.1 Mineral Resource recommendations

Recommendations in respect of the Mineral Resource estimate are as follows:

1. Database validation process needs to continue with improved checks for duplicated RC collars and sub-standard data records.
2. Should pre-2013 RC grade control data be used in future estimates, bias and precision risks will need to be addressed through relevant comparisons with current RC and Diamond data that is supported by robust QAQC results.
3. Current grade control RC sample QAQC must continue in order to maximise value and quality of sample assay results in addition to assuring accurate grade control model estimates and mark out for mining.
4. Vigilance is required with respect to ongoing QA/QC failure criteria and action taken. Annual or six monthly QA/QC reports should be produced to assist with identifying longer term trends such as analytical drift. Umpire (check) assaying should be continued. Some issues were noted with pulp storage and selection and require ongoing monitoring.
5. RC grade control chips should continue to be logged for geology in order to support improved vein volume delineation and resulting estimates.
6. Poor recoveries in solution cavities should be reviewed and procedures adjusted where necessary.
7. Surfaces for the stratigraphic units are treated as sub-horizontal and conformable. However, in-pit mapping is noted for its complex structural deformation, which, if resolved, and at a relevant scale, will need to be considered in future geology models.
8. Knotted schist was domained out of phyllite with the expectation that it was a waste unit. However, mineralisation within that unit shows that either it is not a waste unit, or logging lacks the detail to adequately define this domain. It is recommended that this be reviewed prior to the next MRE update.
9. It is recommended that characteristics of veins be studied so that an improved modelling strategy may be developed for veins that do not have pit exposure. Characteristics such as vein spacing, width and length relative to dome axis could be used to assist and improve the prediction and modelling of veins outside of the pit.
10. It is recommended that the high grade variability at short ranges found in variography be reviewed as part of a spatial grade continuity study to determine reasons for this variability and integrate results into the next Mineral Resource update.
11. Estimation parameters should be reviewed together with the review of mine reconciliation results in order to ensure representative and relevant estimates.

12. It is recommended that vein material that falls outside the vein wireframes be reviewed prior to the next MRE update. It has been accounted for in this MRE through probability modelling and assigning of mean grades to highlight these mineralised zones. However, further work, through either drill testing to increase the dataset, or expansion of the current vein wireframes into other units (i.e. beyond UMC and MMC) is suggested. Any testing of these zones that can take place during mining is also recommended.
13. The vein model was localised to 5m x 5m x 5m. While current mining and production targets allow a higher selectivity around vein material, in future, the SMU will be a consistent 10m x 10m x 10m for all material, which reflects the block size of the block model produced for mine planning. In the next MRE, it is recommended that the new SMU size be reflected in the localisation of the vein model.
14. As with the previous MRE, mineralisation in laterite is not included in the current MRE. This may be attributed to inconsistencies in logging of the stratigraphy and editing of the laterite profile may resolve this. It is recommended that these changes be implemented prior to the next MRE update, so that potential mineralisation is not excluded from the MRE.
15. Contact analysis used to inform the decision of hard/soft boundaries in the estimate was completed between oxidation units. It was found that soft boundaries were appropriate for use, with tight searches defined in the vertical. However, preliminary trend analysis was completed on the grade profile of each domain, and it was found that grade was a function of distance from the dome. A spatial review of the grade profiles towards and away from the dome should be completed to ensure that high grade mineralisation at the domes is sufficiently constrained, and not extrapolated to lower grade areas away from the dome.
16. Improvements have been made in assigning more accurate bulk density and GAC values to blocks, based on additional data collected in 2012 and 2013. However, there are still units that do not have data support. It is recommended that data continue to be collected from these units, in order to more confidently assign accurate values to those blocks.
17. Current reconciliation of "as-mined" volumes between Mineral Resource estimates and Grade Control estimates suggests some opportunity for improving resource and grade control estimates. Primarily, the resource estimates are influenced by much wider drill grid spacing, whereas grade control estimates are influenced by simplistic estimation methods. It is recommended that grade control estimate methods be upgraded to better reflect the methods employed during resource estimation.

26.2 Mineral Reserve recommendations

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

5. A significant aspect of the optimisation and cashflow modelling process, as part of the Mineral Reserve estimate, has been the gathering and review of historical mining and processing costs at KMP and the extrapolation of these in consideration of changing scales of operation. A business intelligence system is being developed at KMP and one of the benefits of this will be in the ability to more efficiently process large amounts of operating data and distil this information into unit costs for mine planning. It is recommended that operating costs continue to be reviewed for future mine planning and that use be made of this system to facilitate the process.

6. Historical costs to date reflect the production and sale of copper cathode and concentrate. Going forward, the Project will sell cathode and anode. Whilst buyers have been contracted through MCT for approximately 70% of the anode copper sales, firm contract pricing is yet to be negotiated. From their commercial experience base, MCT has provided anode metal cost parameters for consideration in the Mineral Reseves estimate for this Technical Report. Whilst, the optimisation sensitivity analyses outlined in this Technical Report have shown minimal impact on marginal cut-off grades when switching from concentrate to anode sales, it is recommended that periodic reviews be undertaken to account for the prevailing and projected market conditions and their impact on future mine planning.
7. Another fundamental aspect of the Mineral Reserves process is the use of variable process recovery relationships, which commensurate with metal price fluctuations, have the most impact on the Mineral Reseve estimation. These relationships therefore need to be periodically reviewed in the light of continuing processing operations, and updates then incorporated into future optimisations and modelling.
8. As part of the normal cyclic development of operating mine plans and mine design updates, the designs shown in this Technical Report should be reviewed against the detailed geotechnical criteria specified by KMP.

ITEM 27 **REFERENCES**

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ITEM 28 CERTIFICATES

David Gray
First Quantum Minerals Ltd
24 Outram St
West Perth, Western Australia, 6005
Tel +61 8 9346 0100 david.gray@FQM.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 "Kansanshi Operations, North West Province, Zambia, NI 43-101 Technical Report" dated effective 31st May 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 Kansanshi property described in the Technical Report in March 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 30th day of June, 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@FQM.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 "Kansanshi Operations, North West Province, Zambia, NI 43-101 Technical Report" dated effective 31st May 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 form 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 Kansanshi property described in the Technical Report in November 2014.

8. I am responsible for the preparation of those portions of the Technical Report relating to Mineral Reserve estimation and Mining, namely Items 15 and 16, respectively. I am also responsible for the preparation of Items 1 to 6, and 19 to 26.

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

10. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement has been in mine planning and the preparation of long term mining plans and production schedules, commencing in 2013.

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

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

Signed and dated this 30th day of June, 2015 at West Perth, Western Australia, Australia.

M. Lawlor

Michael Lawlor

Robert Stone
First Quantum Minerals Ltd
24 Outram St
West Perth, Western Australia, 6005
Tel +61 8 9346 0100 *rob.stone@FQM.com*

I, Robert Stone, do hereby certify that:

1. I am Technical Manager employed by First Quantum Minerals Ltd.

2. This certificate applies to the technical report entitled "Kansanshi Operations, North West Province, Zambia, NI 43-101 Technical Report" dated effective 31st May 2015 (the "Technical Report").

3. I am a professional process engineer having graduated with an undergraduate degree of Bachelor of Science (Honours) from the Camborne School of Mines in 1984.

4. I am a Member of the Institute of Materials, Minerals and Mining (UK). I have been a Chartered Engineer through the Institute of Materials, Minerals and Mining since 1991.

5. I have worked as process engineer and metallurgist for a period in excess of thirty years since my graduation from university. For the last fifteen years I have been in the employ of First Quantum Minerals Ltd in both technical and managerial roles. Of these, seven years were as a manager of process plants producing copper in concentrate, copper as electrowon cathode, gold concentrate and cobalt metal by RLE. The remaining eight years were in a technical role as Consulting Process Metallurgist responsible for development of First Quantum Minerals Ltd projects worldwide including copper/cobalt in Central Africa, nickel in Australia and copper/molybdenum in Panama.

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 Kansanshi property described in the Technical Report in April 2015.

8. I am responsible for the preparation of those items of the Technical Report relating to mineral processing/metallurgical testing and recovery methods, and Project infrastructure, namely Items 13, 17 and 18, respectively.

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 as Plant Manager from 2003 to 2007 covering the design, construction, commissioning and early operations of the Kansanshi Plant. Following this period, my role changed to a technical role, supporting the operations with direct involvement in assisting with upgrade projects. This involved regular interaction with Kansanshi both remotely and by regular visitations

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

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

Signed and dated this 30th day of June, 2015 at West Perth, Western Australia, Australia.

Robert Stone