Kansanshi Operations

North West Province, Zambia

NI 43-101 Technical Report July 2024

Effective Date: 31st December 2023

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DATE AND SIGNATURE PAGE

This NI 43-101 technical report on the Kansanshi Operations in North West Province, Zambia is effective as of 31st December 2023. The report was completed by the following Qualified Persons on behalf of First Quantum Minerals Ltd.

Qualified Person

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Date: 23rd July 2024

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ITEM 1 SUMMARY

This Technical Report on the Kansanshi Operations (the property) has been prepared by Qualified Persons Carmelo Gomez Dominguez, Michael Lawlor and Andrew Briggs of First Quantum Minerals Pty Ltd (FQM, the issuer or the Company). This Technical Report prepared by FQM as an issuer, follows a previous Technical Report filed by FQM in September 2020 and titled *Kansanshi Operations, North West Province, Zambia NI 43-101 Technical Report, June 2020* (the 2020 Technical Report).

The purpose of this particular Technical Report is to document updated Mineral Resource and Mineral Reserve estimates for the property, and to provide a commentary on the status of operations, including a proposed production expansion.

The effective date for the Mineral Resource estimate and the Mineral Reserve estimate is 31st December 2023.

1.1 Mineral Resource and Reserve overview

The FQM September 2020 Measured and Indicated Mineral Resources, as reported in the March 2024 Annual Information Form (AIF, as at the end of December 2023, FQM March 2024), was 812.8 Mt with an average grade of 0.65% TCu (total copper grade). The updated Measured and Indicated Mineral Resource estimate (as at the end of December 2023) now stands at 1,160.9 Mt at an average grade of 0.61 %TCu (excluding stockpiles), and shows an increase of 1.8 Mt of copper metal (+34%) for Measured and Indicated Mineral Resources. The disclosed Mineral Resource estimate includes mining and processing depletions since the end of December 2019 (as per the 2020 Technical Report), as well as additional geological and assaying data obtained from 209 diamond drill holes, 20,561 reverse circulation drill holes and from comprehensive in-pit geological mapping. The increase in Measured and Indicated Mineral Resources is attributable to conversion of Inferred Mineral Resources as well as additions at Main and South East deposits.

Reconciliation from historical production data was used to support and validate the Mineral Resource estimate. Results suggest that plant claimed metal is 5% lower than Mineral Resource estimated metal. This magnitude of risk is aligned with the assigned Mineral Resource classification.

Commensurate with the increase in the Mineral Resource inventory, and also accounting for depletion, the end of December 2023 reported Proven and Probable Mineral Reserve has now risen to 935.2 Mt with an average grade of 0.56%TCu, and with an additional 169.5 Mt stockpiled at an average grade of 0.40%TCu. The insitu copper metal in the combined open pits is 5,266 kt, and with the inclusion of stockpiles is 5,941 kt.

Relative to the mining depleted Mineral Reserve statement in the March 2024 AIF, this reflects an approximate 31% and 19% increase in the Mineral Reserve tonnes and insitu copper within the combined open pits, respectively, mostly attributable to the updated Mineral Resource estimate. As such, the substantially increased Mineral Resource and Mineral Reserve inventories underpin the production expansions planned for Kansanshi.

1.2 Production expansion overview

The current processing rate at Kansanshi is approximately 28 Mtpa from three separate processing circuits treating oxide, transitional/mixed and sulphide plant feed. The 2020 Technical Report described a proposed 25 Mtpa expansion of sulphide processing capacity, with a new stand-alone concentrator (i.e. a third sulphide train) referred to as “S3” commencing in the second half of 2024 and rising to design capacity in the following year.

In line with this proposed expansion was the then scheduled start of mining and sulphide plant feed supply from the South East Dome Pit in the same year. The S3 plant expansion was halted in late 2013 and the part-completed construction work was suspended, along with development plans for mining of the South East Dome Pit.

In May 2022, the Company announced an approval to recommence work on the S3 Project. Continuing through into 2023, the resumed work has focussed on detailed engineering sufficient for issuing purchase orders for long lead time equipment for ore crushing, milling facilities and additional mining fleet. Earthworks and civil works were in progress through 2023 and procurement spending at year end was approximately 70% committed. The S3 sulphide processing capacity has now been increased to a maximum of 27.5 Mtpa.

The S3 plant commissioning is now envisaged during the second quarter of 2025, to be followed by a production ramp-up in Q3 2025. With the Main Pit and the South East Dome Pit providing a substantial contribution to feed tonnes, the S3 plant now has a projected operational life of twenty five years to 2049.

The Kansanshi copper smelter, or KCS, commenced operation in March 2015 with a capacity of 1.2 Mtpa of copper concentrate. This facility currently smelts all of the concentrate from Kansanshi and a portion of the concentrate emanating from Sentinel (i.e. the Company's Trident Project), operating at up to 1.38 Mtpa. Engineering for a phased second smelter was commenced in 2013 but deferred in 2014.

In July 2022, the Company announced approval for expansion of the Kansanshi smelter, increasing throughput capacity to 1.6 Mtpa. Engineering on the smelter expansion has proceeded with orders placed for long lead items associated with the oxygen plant, acid plant, and wet electrostatic precipitation.

1.3 Mining overview

Mining of the Main Pit is currently scheduled to continue until 2046, whereas mining of the North West Pit is scheduled for completion by 2040. Bush clearing and topsoil mining were eventually commenced at South East Dome in December 2023, whilst waste pre-stripping is currently scheduled to commence during the current year.

The first new ultra class trucks and first new shovel have been commissioned and are operating in the South East Dome Pit.

1.4 Project location and ownership

The Kansanshi Operations are located approximately 10 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 Copper belt, is approximately 180 km to the southeast of Solwezi.

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

1.5 Project background

Mining at Kansanshi is currently carried out in two open pits, the Main and North West Pits, using conventional open pit methods, with electric and hydraulic excavators together with a mixed fleet of haul trucks. Since 2014, total material movement from the Main and North West Pits has been in the range of 80 to 110 Mtpa.

A nearby deposit has been defined at South East Dome, the primary ore mineralisation from which will partially sustain the proposed expansion of sulphide ore processing. Initial site clearing and material movement has commenced.

Ore processing is flexible to allow for variation in plant feed either through a solvent extraction and 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 various cupriferous ore types 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 first stage on-site smelter, the construction of which was completed in the same year. This allowed ore previously classified as mixed to be leached, thereby reducing the plant feed types to essentially two, i.e. float/leach and sulphide. The current treatment capacity at Kansanshi is approximately 15 Mtpa of float/leach feed and approximately 13 Mtpa of sulphide feed.

The KCS facility was commissioned in late 2014, with a nameplate 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 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.

Rather than constructing a second smelter, expansion of the existing smelter to treat up to 1.6 Mtpa of concentrate is currently in progress, and will continue in 2024. Final tie-ins for new equipment will be implemented during a planned smelter shut-down in June 2025.

Further capital works also commenced in early 2013 to increase the sulphide flotation capacity to around 25 Mtpa (the S3 expansion, but with a revised capacity of 27.5 Mtpa). Mine planning and Mineral Reserves estimation/reporting in the NI 43-101 Technical Report filed by FQM in May 2015 and titled *Kansanshi Operations, North West Province, Zambia, NI 43-101 Technical Report* (the 2015 Technical Report) documented the S3 expansion commencing in mid-2017, with contributory plant feed coming from the South East Dome Pit in the same year. Development progress on S3 had been placed on hold in late 2013.

This Technical Report addresses the resumed construction of the S3 Project, with commissioning now commencing in Q2 2025, and with production ramp-up then continuing through 2025 and ultimately reaching a revised rate of 27.5 Mtpa. Pre-strip mining at the South East Dome Pit has commenced with first ore feed to the S3 plant now proposed for 2025, and with feed contribution principally from Main Pit cutbacks continuing through to 2049.

Figure 1-1 Is a plan of the Kansanshi Operations showing the extent of current mine operational areas (i.e. Main Pit and North West Pit) and the location of the design South East Dome Pit. Also shown on Figure 1-1 are the perimeter extents of the Mineral Reserve ultimate pit (i.e. shown as a green line), plus locations of the existing processing plant, the S3 plant, the smelter and the tailings storage facilities.

It can be appreciated from this figure that longer term pit cutbacks will involve the part-dismantling of existing process plant infrastructure. This will occur following the currently scheduled cessation of oxide, mixed and S2 processing in 2039.

1.5.1 Project tenure

All surface rights necessary for the Kansanshi Operations have been obtained and include six leases within the mining licence covering in excess of 9,500 ha. The right to mine is governed by a large-scale mining licence (7057-HQ-LML, formerly LML 16) granted in March 1997, which had a term of 25 years (to 2022). In February 2022 the mining licence was renewed for a further 25 years until March 2047. Conditions of grant of the renewed licence remain unchanged.

In December 2000, the licence area was increased from 4,244 ha to 21,593 ha, and in November 2014 the area was further increased to 24,865 ha (Figure 1-2).

Figure 1-1 Kansanshi Operations site plan, December 2023 (source: FQM)

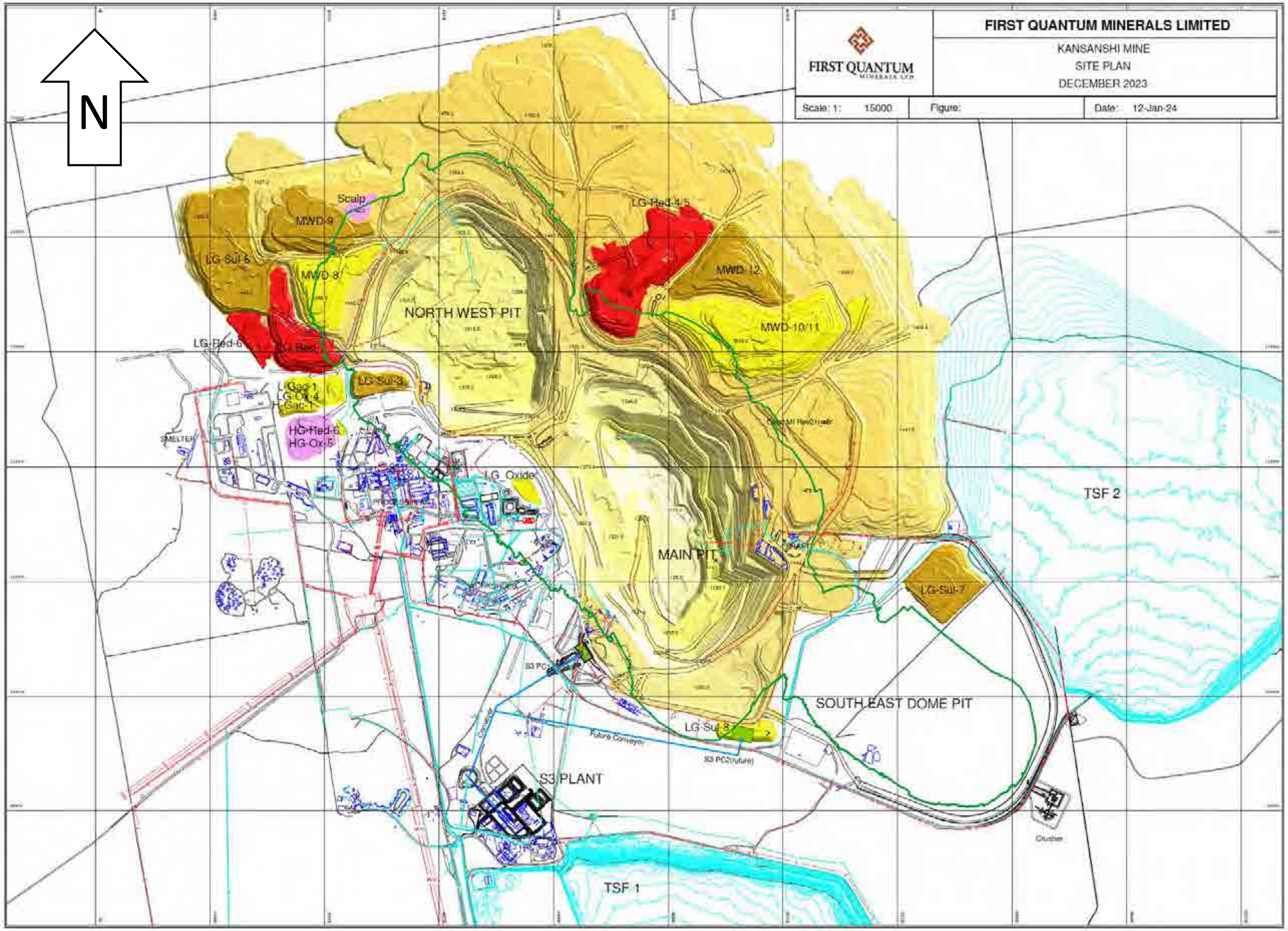
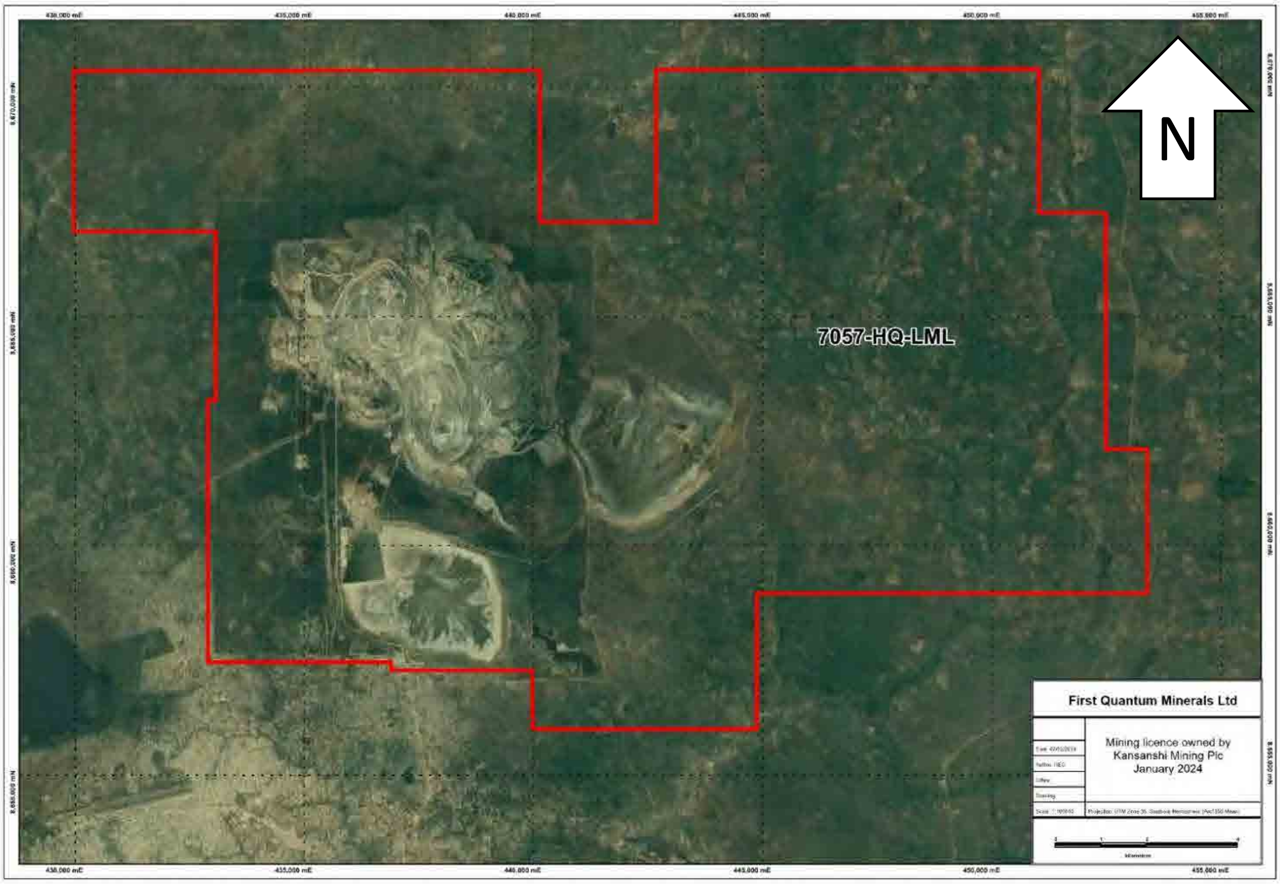


Figure 1-2 Extents of Large-Scale Mining Licence 7057-HQ-LML, December 2023 (source: FQM)



1.5.2 Geology and mineralisation

Regionally, the Kansanshi deposits (Main, North West and South East) are located within the deformed metasediments of the Nguba (formerly Lower Kundulungu) Group, which is part of the Katanga Supergroup in the Zambian Copperbelt. Locally, the deposits are situated within domal structures along the crest of a regional antiform. Deposit mineralisation is closely associated with these domes and is localised to a structurally modified sequence of rock units comprised of dolomites, dolomitic marbles, various schists and phyllites. Until mid-2023, mining had been restricted to the Main and North West deposits.

The dominant primary sulphide copper mineralisation and geology may be summarised as follows:

- stratabound with mostly disseminated and veinlet style mineralisation
- sub-vertically dipping, quartz-carbonate-sulphide veins that crosscut stratigraphy, and
- localised brecciated style of mineralisation

The primary sulphide mineralisation is influenced by weathering and oxidation with:

- near surface weathering in the saprolitic zone resulting in residual copper styles of mineralisation
- around vertical veins, with oxide copper mineralisation forming, such as malachite, tenorite and chrysocolla
- transitional weathering zones with mixed primary and secondary copper sulphide copper mineral assemblages
- pervasive shallow to deep weathering located along geological structures

Primary sulphide copper mineralisation is mostly chalcopyrite, with minor bornite. Oxide mineralisation is mostly chrysocolla with malachite. The transition zone contains mixed copper oxides, primary copper sulphides, secondary copper sulphides and minor native copper and tenorite. Minor copper is hosted in clay and mica minerals, and is classified as refractory. Gold is generally positively associated with copper mineralisation.

1.5.3 Exploration status

Since the 2020 Technical Report, FQM has commissioned an additional airborne electromagnetic geophysical survey which targets new domal structures located along strike and adjacent to the Kansanshi dome. Exploration work has included eleven diamond holes drilled during 2023.

1.5.4 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, transitional/mixed, and oxide ores independently. The flexibility of the circuitry, however, allows these ore types 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 circuit was subject to an upgrade in 2014; effectively doubling the leach capacity. This expansion has matched the output of low-cost acid from the Kansanshi smelter and allows ore previously classified as mixed to be leached, thereby reducing the plant feed types to essentially two, i.e. float/leach and sulphide.

The S3 plant expansion will be a stand-alone sulphide ore concentrator that does not impact upon the operation of the remainder of the Kansanshi processing plant.

1.5.5 Production performance

As a comparison between the planned production as described in the 2020 Technical Report, and the actual performance for the period 2021 to the end of 2023:

- ore and waste tonnes mined were lower
- direct feed tonnage was lower, whilst stockpile reclaim tonnage was higher
- total plant feed tonnage was 91% of that projected
- the lesser tonnage was offset by a higher average feed grade than that projected
- the insitu copper metal in the overall plant feed was 3% less than projected

1.5.6 S3 Project development status

The S3 Project is a 27.5 Mtpa sulphide copper concentrator that was originally approved for development in December 2012. Construction was suspended in 2013, except for specific works completed through into 2015, as follows:

- completion of the engineering for most disciplines, except for electrical design
- completion and commissioning of a concentrate filtration plant, which has been in continuous operation providing feed to the KCS since October 2015
- completion of the mill building concrete and structural steel
- completion of the stockpile reclaim vault concrete
- completion of the majority of the process plant concrete, particularly for the flotation and thickening circuit

The work programme for recommencement of the S3 Project includes:

- resumption and completion of remaining engineering design work, including a revision to the flotation design circuit to incorporate controlled potential sulphidisation (CPS) capability catering for oxidised ores and secondary sulphide, and additional concentrate cleaning capacity
- redesign of the flotation circuit to allow for cleaner flotation within the existing processing facilities at Kansanshi
- review of the revised mine plan and an update on the engineering of the primary crushers and overland conveyor infrastructure
- recommencement of procurement for all outstanding construction items
- procurement of the additional mining fleet to support expanded production

The S3 Project is scheduled for commissioning in Q2 2025 with production ramp up during Q3 2025.

1.5.7 New cleaner flotation circuit

The S3 cleaning circuit has been partially constructed at the existing plant to support early enhancement of sulphide concentrate grade and to provide depressing facilities for carbon and pyrite in order to support enhanced smelter treatment rates. This cleaning circuit was commissioned in Q4 2021 and comprises two column cells, two Jameson cells and a bank of conventional flotation cells.

The cleaning circuit will be expanded to full S3 capacity with the installation of two additional column cells, two Concorde cells and a bank of larger flotation cells.

1.5.8 Smelter expansion

The KCS has performed well since commissioning and is currently operating at a capacity of up to 1.38 Mtpa of copper concentrate (i.e., 15% above its nameplate capacity) sourced from the existing Kansanshi circuits and delivered from Sentinel. This capacity has been achieved through optimisation and de-bottlenecking efforts. Commissioning of the S3 circuit along with proposed increases in throughput from the Company's Sentinel plant[1], will result in a varying future concentrate feed rate. This increased concentrate feed will be achieved and accommodated by:

- an increase in smelter throughput, up to 1.6 Mtpa
- 0.2 Mtpa treated through the existing HPL circuit, boosted by an increased supply of high pressure oxygen
- as much as 0.4 Mtpa marketed to other Zambian smelters

The expanded smelter capacity will allow consideration of purchased concentrate to enable a sustained 1.6 Mtpa throughput to be achieved.

The smelter capacity increase will be achieved through:

- improvements in concentrate cleaning at both Kansanshi and Sentinel, leading to the delivery of higher grade concentrates containing lower levels of carbon and pyrite
- increased oxygen addition to the smelter
- additional cooling facilities on the Isasmelt furnace

[1] A proposed expansion of the Sentinel processing facilities to 62 Mtpa throughput is reported in an NI 43-101 Technical Report on the Trident Project, dated March 2020 (FQM, March 2020).

- inclusion of an Isoconvert furnace, taking the place of an additional Pierce-Smith converter (this has already been installed)
- additional acid plant capacity by converting a redundant sulphur burning acid plant at Kansanshi to handle the additional smelter off-gasses produced

This work is currently in progress and is expected to be completed with final tie-ins during a planned smelter shut-down in June 2025.

1.5.9 Reduction of greenhouse gas emissions

Several initiatives are underway at Kansanshi as a commitment to reducing greenhouse gas emissions. Whilst trolley assisted haulage has been a feature of the mine for some time,

- assisted haulage routes are being extended onto flat ramp segments beyond the pit rim (i.e., yielding a further reduction in diesel fuel consumption), and
- battery electric haul trucks will be trialled in 2024 as a means of further reducing diesel fuel consumption (i.e., the truck batteries can be recharged whilst proceeding under trolley assist)

The Company is also working with ZESCO, the Zambian power supply provider, to transition to additional renewable power sources other than the current hydroelectric source. This includes solar and wind turbine generated power.

1.5.10 Environmental approvals and status

Twenty-five Environmental Impact Assessments (EIAs) for operational infrastructure at KMP have been submitted and approved by the Zambia Environmental Management Agency (ZEMA) over the last eighteen 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 Performance Standards, in addition to the World Bank EHS Guidelines for Mining.

The Company is implementing an Environmental Management System based on the ISO 1400:2015 Standard. The Standard provides 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 December 2023 and the Company has not been subject to any penalties arising because 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.

1.6 Changes since the 2020 Technical Report

Noteworthy changes from the 2020 Technical Report information are:

- a now proposed S3 commissioning date of Q2 2025, with capacity increased from 25 Mtpa in 2026 to 27.5 Mtpa from 2032
- the existing KCS facility is being increased in capacity to up to 1.6 Mtpa
- additional concentrate treatment capacity will be achieved through the existing HPL circuit
- commencement of waste pre-stripping at South East Dome in 2024
- a significant uplift in the Mineral Reserve, largely attributed to:

  - a 43% increase in the Measured and Indicated Mineral Resource tonnes together with a commensurate 6% decrease in the total copper grade. Estimate changes are due to significant additions in drill sample data as well as added detail to the vein style mineralisation
  - Inferred Mineral Resource tonnages have reduced by 70% due to reclassification into the Measured and Indicated categories
- relative to the 2020 life of mine production schedule, the mine life has been extended to 2046, with processing to 2049
- incorporation into mine planning models of modelled mining dilution and loss adjustments relating to varying styles of mineralisation each with an associated metal (loss) risk assessment (i.e. an aspect that was not previously considered)
- updated mining unit cost estimates
- updated processing and general and administration (G&A) unit cost estimates for all existing circuits and for the proposed S3 circuit
- updated variable processing recovery relationships drawing on operational performance trends

Capital costs have been re-estimated to account for the resumption of and current status of S3 expansion works and these costs are included in a supporting Mineral Reserve cashflow model.

1.6.1 Mineral Resource summary

The Mineral Resource estimate update was completed by Qualified Person Carmelo Gomez of FQM, with the assistance of Kansanshi mine geologists.

This Mineral Resource estimate update includes data from 2,179 diamond drilled holes (DD) and 73,353 reverse circulation (RC) drilled holes. Since the 2020 Technical Report, data from 209 DD holes and 20,561 RC holes has been added. The drill hole data (logging and sampling), combined with in-pit geology mapping has facilitated improved geological modelling of vein volumes, stratigraphy and the interpretation of weathering and oxidation profiles, resulting in improved accuracy of the estimates and supporting upgrades to the Mineral Resources classification. The Mineral Resource estimates show an increase of 1.8 Mt of copper metal for Measured and Indicated.

Copper (Cu), Acid soluble copper (ASCu), and gold (Au) grade estimates were completed in three stages:

1. Ordinary kriging was used to estimate grades within the 3D wireframe modelled veins.
2. A categorical indicator was used to define mineralised and non-mineralised strata, followed by ordinary kriging with dynamic anisotropy to estimate metal grades within the strata.
3. In areas with wider drill grid spacing, non-wireframed veins were estimated by assigning a proportion of a block to vein material. Vein true widths were used to estimate vein volumes, and ordinary kriged grades were assigned to these vein volumes.

Block model grade estimates were validated visually, with summary statistics, swath plots, by comparison with previous estimates, and against mine reconciliation data. These validations support the estimates as representative of their input sample data and the prevailing geology.

Estimates were classified as Measured, Indicated and Inferred Mineral Resources according to confidence in the prevailing geology, the estimation methods used, and the resulting estimated grades. Drill grid spacing, QAQC (quality assurance / quality control), and grade estimate confidences were each considered for the assigned classification. An ultimate pit shell was used to guide the limits of mineralisation in order to ensure there are reasonable prospects for eventual economic extraction. Laterite and refractory mineralisation, which are not recoverable, were not classified.

The estimated and classified Kansanshi Mineral Resource statement is presented in Table 1-1. The estimate is reported at a 0.2% total copper cut-off grade, consistent with the cut-off grade applied to the Mineral Reserve estimate. Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Since the 2020 Technical Report, this estimate accounts for mining depletion and includes added drill hole data. The estimate benefits from improved geology models and estimation routines.

In addition to the unmined Mineral Resources, several stockpiles were generated during open pit mining operation. Current stockpiles include separate areas for oxide, mixed and sulphide materials (Table 1-2).

Table 1-1 Kansanshi Mineral Resource statement, excluding stockpiles, at a 0.2% Total Copper cut-off and depleted of mined material as at 31st December 2023

Classification	Volume (millions)	Density (t/m^3)	Tonnes (millions)	Cu (1) (%)	AsCu (%)	Au (g/t)	Cu metal (1) (kt)	AsCu metal (kt)	Au metal (2) (koz)
Measured	164.1	2.57	422.0	0.68	0.12	0.12	2,881.0	520.3	1,674.2
Indicated	270.6	2.73	738.9	0.57	0.06	0.12	4,244.5	429.0	2,811.2
Total Measured & Indicated	**434.7**	**2.67**	**1,160.9**	**0.61**	**0.08**	**0.12**	**7,125.5**	**949.4**	**4,485.5**
Inferred	17.9	2.75	49.3	0.41	0.02	0.09	200.2	11.1	143.9

(1) Cu (%) grade is inclusive of AsCu (%) grade. Cu metal (kt) is inclusive of AsCu metal (kt).

(2) 1 troy ounce= 31.1035 grams

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

Table 1-2 Kansanshi Mineral Resource statement for stockpiles as at 31st December 2023

Stockpiles	Classification	Tonnes (Mt)	Cu (1) (%)	AsCu (%)
Oxide (float/leach feed)	Indicated	54.1	0.30	0.11
Mixed (float/leach feed)		45.7	0.57	0.18
Sulphide (float feed)		69.7	0.36	0.01
Total Stockpile - Indicated		**169.5**	**0.40**	**0.09**

(1) Cu (%) grade is inclusive of AsCu (%) grade. Cu metal (kt) is inclusive of AsCu metal (kt).

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

1.6.2 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 with overview by Qualified Person Michael Lawlor of FQM.

Conventional Whittle Four-X software was used as a guide in refining the cutbacks and ultimate extents of the Main and North West pits, and also for the South East Dome Pit. The optimisation process considered pit slope design criteria provided by KMP geotechnical staff, in addition to mining/processing operating costs derived and extrapolated from actual costs and performance indicators.

Pit designs were guided by a selected optimised pit shell, with adaptations to mining phases relating to the current mining areas (as at December 2023) and considering constraints imposed by existing infrastructure. 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 1-3. The additional Mineral Reserve within stockpiles (as at December 2023) is stated in Table 1-4.

An important aspect of the production scheduling and Mineral Reserve estimation was the detailed consideration of both planned and unplanned mining losses. The methodology for determining the quantum of each of these modifying factors was through the application of a model reblocking and smoothing process to emulate the mining of practically mineable parcels. This application was a development and enhancement of the process adopted and described in the 2020 Technical Report, in so far as metal loss risk was assessed against the varying mineralisation styles in each deposit. In addition, production tracking records were taken into account, as were the site-based metal loss projections for the near-term, five-year planning period.

To support the Mineral Reserve estimate, a cashflow model has been prepared, inclusive of updated operating and metal (TCRC and royalty) costs. This model includes estimated capital and sustaining expenditure, and is presented pre-tax and post-tax.

The total Mineral Reserve for the Kansanshi Operations, inclusive of existing stockpiles, is **1,104.8 Mt at an average grade of 0.54 %TCu**. The reported Mineral Reserve is based on an economic copper equivalent cut-off grade which accounts for longer-term copper and gold price projections of $3.50/lb ($7,717/t) and $1,805/oz, respectively.

The depleted Mineral Reserve as reported in the Company's Annual Information Form (AIF) at the end of December 2023 (FQM, March 2024), and relative to the Mineral Resource model in use for the 2020 Technical Report, was 715.3 Mt at an average grade of 0.62 %TCu (i.e., excluding stockpiles).

The largest contributor to the increase in the Mineral Reserve tonnage inventory is due to the updated Mineral Resource model. The reduction in overall average copper grade, between the AIF report and this latest Mineral Reserve estimate, can be attributed largely to the reduction in the overall estimated average grade in the Mineral Resource model (specifically for the Main and North West deposits). Further grade reductions have come about due to the incorporation of planned and unplanned mining dilution (i.e., metal losses).

Table 1-3 Kansanshi Pit Mineral Reserve statement, at 31st December 2023

MINERAL RESERVE AT 31st DECEMBER 2023 (AT $3.50/lb Cu AND $1805/ounce Au)					
Oxide Ore					
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
MAIN & NW	Proven	118.7	0.53	0.25	0.09
	Probable	60.0	0.52	0.23	0.11
	Total P+P	**178.7**	**0.52**	**0.24**	**0.10**
SE DOME	Proven	15.3	0.43	0.15	0.07
	Probable	16.5	0.35	0.12	0.06
	Total P+P	**31.9**	**0.39**	**0.13**	**0.07**
TOTAL	**Proven**	**134.1**	**0.51**	**0.24**	**0.09**
	Probable	**76.5**	**0.48**	**0.21**	**0.10**
	Total P+P	**210.6**	**0.50**	**0.23**	**0.09**
Mixed Ore					
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
MAIN & NW	Proven	41.6	0.86	0.23	0.18
	Probable	29.1	0.86	0.22	0.16
	Total P+P	**70.6**	**0.86**	**0.22**	**0.17**
SE DOME	Proven	2.9	0.54	0.14	0.09
	Probable	8.0	0.66	0.17	0.10
	Total P+P	**10.9**	**0.63**	**0.16**	**0.09**
TOTAL	**Proven**	**44.4**	**0.84**	**0.22**	**0.17**
	Probable	**37.1**	**0.82**	**0.21**	**0.15**
	Total P+P	**81.5**	**0.83**	**0.21**	**0.16**
Sulphide Ore					
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
MAIN & NW	Proven	175.6	0.59	0.02	0.12
	Probable	297.0	0.51	0.02	0.12
	Total P+P	**472.6**	**0.54**	**0.02**	**0.12**
SE DOME	Proven	86.1	0.62	0.02	0.12
	Probable	84.4	0.51	0.01	0.09
	Total P+P	**170.5**	**0.57**	**0.02**	**0.11**
TOTAL	**Proven**	**261.7**	**0.60**	**0.02**	**0.12**
	Probable	**381.4**	**0.51**	**0.02**	**0.11**
	Total P+P	**643.1**	**0.55**	**0.02**	**0.12**
Total Ore					
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
MAIN & NW	Proven	335.9	0.60	0.13	0.12
	Probable	386.1	0.54	0.07	0.12
	Total P+P	**722.0**	**0.57**	**0.10**	**0.12**
SE DOME	Proven	104.3	0.59	0.04	0.11
	Probable	109.0	0.50	0.04	0.09
	Total P+P	**213.2**	**0.54**	**0.04**	**0.10**
TOTAL	**Proven**	**440.2**	**0.60**	**0.11**	**0.12**
	Probable	**495.0**	**0.53**	**0.06**	**0.11**
	Total P+P	**935.2**	**0.56**	**0.08**	**0.11**

Table 1-4 Kansanshi Stockpile Mineral Reserve statement, at 31st December 2023

MINERAL RESERVE AT 31st DECEMBER 2023 (AT $3.50/lb Cu AND $1805/ounce Au)					
Oxide Ore					
S/piles	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
TOTAL	Proven	-	-	-	-
	Probable	54.1	0.30	0.11	-
	Total P+P	**54.1**	**0.30**	**0.11**	**-**
Mixed Ore					
S/piles	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
TOTAL	Proven	-	-	-	-
	Probable	45.7	0.57	0.18	-
	Total P+P	**45.7**	**0.57**	**0.18**	**-**
Sulphide Ore					
S/piles	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
TOTAL	Proven	-	-	-	-
	Probable	69.7	0.36	0.01	-
	Total P+P	**69.7**	**0.36**	**0.01**	**-**
Total Ore					
S/piles	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
TOTAL	**Proven**	**-**	**-**	**-**	**-**
	Probable	**169.5**	**0.40**	**0.09**	**-**
	Total P+P	**169.5**	**0.40**	**0.09**	**-**

1.6.3 Production schedule

Table 1-5 and Table 1-6 summarise the LOM production schedule, whilst Figure 1-3 shows the LOM mining sequence. Features of the LOM mining and production schedule associated with the detailed pit designs are as follows:

1. The total material mined from all pits amounts to 3,653.6 Mt (1,420.8 Mbcm), of which 935.2 Mt is ore (including oxide, mixed and sulphide ore) and 2,718.3 Mt is waste (including refractory and Inferred Resource). This is after the application of planned and unplanned mining losses.
2. Mill feed rates in the first years of the schedule have been optimised upon the feed availability, and as such, vary slightly each year. From 2028 and for the remainder of the life of mine, the mill feed capacities are set at 7.3 Mtpa for oxide ore feed, 7.8 Mtpa for mixed ore feed, 13.5 Mtpa for the sulphide ore into the existing S2 plant and up to an additional 27.5 Mtpa for the S3 plant. The exceptions to these set capacities are:
   - in 2027, when the mixed feed plant processes 12.4 Mt to compensate for a lack of S2 plant feed
   - and again in 2028 and 2039 for the same reason, when the mixed feed plant processes 13.9 Mt in each year
3. The S3 plant will start up in Q2 2025 (13.9 Mt processed) and be fully operational in 2026.
4. The LOM mining sequence is driven largely by opening up additional feed areas for the S3 expansion.
5. Annual mined volume ramps up to 80 million bcm for a period between 2024 and 2031, and remains at that level until 2038, thereafter trending downwards until the end of mining in 2046 (Figure 1-3).
6. The majority of additional material being mined is characterised as near surface overburden waste material, which can be bulk mined efficiently on wide terraced pushbacks.

7. In the mine sequencing, the higher strip ratio pushbacks are delayed towards the end of the schedule to limit the annual mining volumes.
8. Main Pit is mined extensively throughout the life of the mine and has three main phases (m16, m21 and m28). It is completed in 2046, three years ahead of processing completion.
9. North West Pit is mined and depleted by 2040 to open up space for waste backfill on the north wall.
10. Pre-strip mining of South East Dome began in 2024 and the pit is scheduled for completion in 2039.

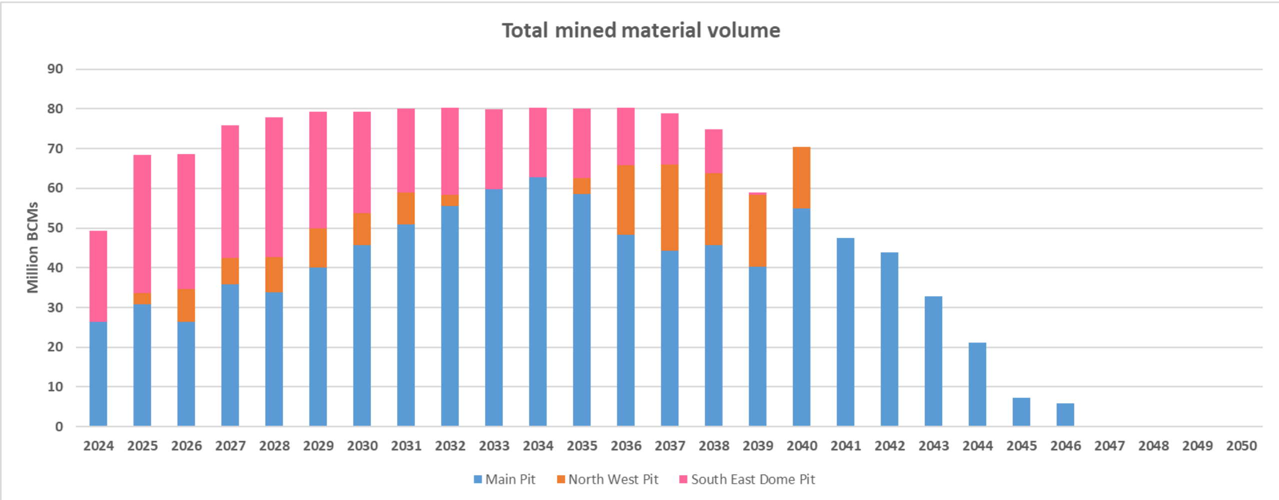
Table 1-5 Kansanshi LOM schedule, ore and waste mining (after planned and unplanned mining losses)

Year	Mined Ore								Total Mined (Mt)	Mined Waste (Mt)	Strip ratio
	Oxide (Mt)	Mixed (Mt)	Sulphide (Mt)	Main Pit (Mt)	NW Pit (Mt)	SED Pit (Mt)	Total (Mt)	Grade (%TCu)			
2024	8.7	5.6	13.4	27.3	0.0	0.4	27.7	0.74	103.5	75.8	2.7
2025	17.4	7.7	20.9	31.6	4.3	10.2	46.1	0.71	153.9	107.9	2.3
2026	20.8	5.6	19.0	27.5	1.4	16.5	45.4	0.55	161.7	116.3	2.6
2027	14.4	4.9	24.9	19.7	0.9	23.6	44.2	0.59	178.6	134.4	3.0
2028	19.7	3.5	26.7	18.3	5.6	26.1	49.9	0.55	188.9	139.0	2.8
2029	22.6	4.8	25.9	28.8	10.6	14.0	53.4	0.56	206.7	153.4	2.9
2030	15.3	7.0	39.2	37.3	8.1	16.2	61.6	0.57	209.2	147.6	2.4
2031	11.9	6.0	36.1	30.0	8.5	15.5	53.9	0.61	211.8	157.8	2.9
2032	8.8	5.9	37.6	37.9	2.6	11.8	52.3	0.60	206.4	154.1	2.9
2033	7.0	5.9	45.0	44.2	0.0	13.8	57.9	0.58	209.4	151.4	2.6
2034	7.0	4.8	48.6	45.1	0.0	15.2	60.3	0.52	222.1	161.8	2.7
2035	9.1	5.4	44.5	40.6	0.0	18.4	59.0	0.53	217.8	158.7	2.7
2036	10.2	4.4	37.5	38.5	0.1	13.6	52.1	0.55	203.7	151.6	2.9
2037	9.9	3.7	31.6	36.7	0.6	7.8	45.1	0.55	203.2	158.1	3.5
2038	10.6	2.4	35.0	29.9	8.0	10.1	48.0	0.55	197.3	149.4	3.1
2039	9.1	2.4	22.1	19.8	13.4	0.4	33.5	0.57	161.6	128.1	3.8
2040	2.3	0.8	28.9	17.0	15.0		32.0	0.51	176.9	144.9	4.5
2041	1.5	0.3	15.0	16.8			16.8	0.52	130.1	113.3	6.8
2042	0.6	0.1	23.0	23.7			23.7	0.58	122.5	98.8	4.2
2043	1.9	0.2	26.5	28.7			28.7	0.47	91.8	63.1	2.2
2044	1.7	0.2	19.6	21.6			21.6	0.44	59.3	37.7	1.7
2045			11.4	11.4			11.4	0.35	20.6	9.2	0.8
2046			10.6	10.6			10.6	0.31	16.5	5.9	0.6
2047											
2048											
2049											
2050											
TOTAL	**210.6**	**81.5**	**643.1**	**643.0**	**79.0**	**213.2**	**935.2**	**0.56**	**3,653.6**	**2,718.3**	**2.9**

Table 1-6 Kansanshi LOM schedule, plant feed by circuit (after planned and unplanned mining losses)

Year	Direct Feed	Reclaim Feed	Total Feed			Insitu metal		Rec'd metal		Overall recovery (%)	
	(Mt)	(Mt)	(Mt)	(%TCu)	(g/t Au)	(kt Cu)	(koz Au)	(kt Cu)	(koz Au)	(Cu)	(Au)
2024	13.0	15.7	28.7	0.68	0.16	195.9	151.0	157.6	49.2	80.5%	32.6%
2025	21.8	20.1	42.5	0.62	0.15	263.4	211.1	215.1	70.7	81.7%	33.5%
2026	26.5	26.5	53.5	0.49	0.11	260.0	183.2	213.7	62.4	82.2%	34.1%
2027	33.8	19.5	53.3	0.56	0.11	296.5	191.4	241.9	64.5	81.6%	33.7%
2028	33.9	19.7	53.7	0.55	0.10	297.4	178.5	248.9	62.4	83.7%	35.0%
2029	36.5	17.1	53.6	0.55	0.10	293.8	180.2	242.2	62.2	82.4%	34.5%
2030	48.3	5.3	53.6	0.61	0.11	327.6	192.5	270.8	66.5	82.7%	34.6%
2031	47.8	5.7	53.6	0.61	0.11	329.2	182.2	273.8	62.9	83.2%	34.5%
2032	49.3	6.9	56.1	0.58	0.11	327.9	207.1	274.6	72.1	83.7%	34.8%
2033	52.1	3.9	56.0	0.59	0.12	331.3	211.0	276.8	73.7	83.5%	34.9%
2034	50.5	5.4	56.0	0.55	0.10	306.4	179.8	259.3	63.4	84.6%	35.3%
2035	51.4	4.6	56.0	0.55	0.10	309.1	184.1	261.4	64.8	84.6%	35.2%
2036	47.2	8.9	56.1	0.54	0.10	302.3	174.1	253.6	61.0	83.9%	35.0%
2037	40.1	15.8	56.0	0.51	0.10	287.6	185.4	240.5	65.1	83.6%	35.1%
2038	46.2	9.6	55.7	0.55	0.11	304.2	199.0	249.6	69.2	82.0%	34.8%
2039	33.5	22.2	55.7	0.47	0.10	264.2	172.2	211.3	59.1	80.0%	34.3%
2040	27.4	0.0	27.5	0.55	0.12	150.5	102.1	135.6	38.3	90.1%	37.5%
2041	15.5	11.9	27.4	0.44	0.10	119.7	90.5	102.3	33.9	85.5%	37.5%
2042	23.2	4.2	27.4	0.54	0.11	149.1	99.9	133.5	37.5	89.5%	37.5%
2043	27.0	0.4	27.4	0.47	0.10	130.0	91.9	117.2	34.5	90.2%	37.5%
2044	20.2	7.3	27.4	0.41	0.09	111.8	79.2	96.0	29.7	85.8%	37.5%
2045	11.4	15.9	27.4	0.31	0.09	85.7	77.9	68.4	29.2	79.8%	37.5%
2046	10.6	16.8	27.4	0.31	0.10	84.5	87.8	65.8	32.9	77.9%	37.5%
2047		27.4	27.4	0.33	0.07	91.3	58.5	62.5	21.9	68.4%	37.5%
2048		27.5	27.5	0.33	0.07	91.6	58.6	62.6	22.0	68.4%	37.5%
2049		19.5	18.1	0.33	0.07	60.2	38.6	41.2	14.5	68.4%	37.5%
2050											
TOTAL	**767.2**	**337.6**	**1,104.7**	**0.52**	**0.11**	**5,771.1**	**3,767.7**	**4,776.3**	**1,323.7**	**82.8%**	**35.1%**

Figure 1-3 Kansanshi life of mine schedule, mining sequence



In order to consider infrastructure and longer term equipment requirements, a production schedule was also completed inclusive of Inferred Resource, mined and processed after depletion of the Mineral Reserve inventory. If this Inferred Resource was able to be converted to a Mineral Reserve, this indicative schedule shows that notionally, the mine life could be extended to 2049 with processing to 2053.

1.6.4 Capital and sustaining cost estimates

Table 1-7 lists the updated capital cost estimates related to the S3 plant and smelter expansions, in addition to the projected costs for additional mining equipment, and other expenses associated with the expansion.

The three largest items, i.e. the S3 expansion, the additional mining fleet and the South East Dome pre-strip, are projected to cost a total of $1.25 B inclusive of amounts spent in the preceding two years.

With respect to the S3 expansion total projected cost of $775 M, this figure can be compared with the $408 M projected cost as reported in the 2020 Technical Report. The increase can be attributed to design changes, inclusion of additional facilities and inflation related costs. The increased total projected cost of $115 M for the smelter upgrade, relative to the $80 M estimate reported in 2020, can be attributed also to the inclusion of additional facilities and inflation related costs.

Table 1-7 Kansanshi capital cost provisions

TOTAL	BEFORE 2024	FROM 2024	TOTAL	2024	2025	2026	2027	2028	2029	2030
Mining capital ($M)										
S3 Mining fleet	**$63.0**	**$178.3**	**$241.3**	$178.3						
subtotal ($M)	**$63.0**	**$178.3**	**$241.3**	**$178.3**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**
Processing capital ($M)										
S3 expansion	**$151.1**	**$623.9**	**$775.0**	$429.5	$152.5	$41.8				
Smelter expansion / upgrade	**$40.0**	**$75.0**	**$115.0**	$47.0	$28.0					
Process plant upgrade	**$0.0**	**$13.2**	**$13.2**	$3.9	$8.7	$0.6				
TSF expansion	**$0.0**	**$116.0**	**$116.0**	$23.7	$17.9	$38.5	$23.9	$11.8		
Other site infrastructure and ancillary equipment	**$0.0**	**$27.8**	**$27.8**	$9.8	$6.5	$3.9	$4.0	$3.5		
TSF 3	**$0.0**	**$94.0**	**$94.0**					$25.0	$69.0	
subtotal ($M)	**$191.1**	**$949.8**	**$1,140.9**	**$513.9**	**$213.7**	**$84.9**	**$27.9**	**$40.3**	**$69.0**	**$0.0**
Other capital ($M)										
SE Dome pre-strip	**$0.0**	**$236.0**	**$236.0**	$68.6	$121.5	$45.9				
Other stripping	**$0.0**	**$182.9**	**$182.9**			$61.5	$79.0	$42.4		
Dewatering shaft & associated development	**$0.0**	**$21.6**	**$21.6**	$21.6						
Trolley line Installations	**$0.0**	**$30.7**	**$30.7**	$11.6	$10.9	$3.8	$4.3			
Second primary crusher	**$0.0**	**$70.0**	**$70.0**		$5.0	$20.0	$45.0			
Gravity gold	**$0.3**	**$0.3**	**$0.7**	$0.3						
Other (aggregate crusher, HPL oxygen upgrade, and other)	**$0.0**	**$18.5**	**$18.5**	$17.5	$1.0					
subtotal ($M)	**$0.3**	**$560.0**	**$560.3**	**$119.7**	**$138.4**	**$131.3**	**$128.3**	**$42.4**	**$0.0**	**$0.0**
Total ($M)	**$254.4**	**$1,688.0**	**$1,942.5**	**$811.9**	**$352.0**	**$216.2**	**$156.2**	**$82.7**	**$69.0**	**$0.0**

In relation to the S3 processing capital cost listed in Table 1-7, the figure of $775 M covers:

- $480 M for direct costs inclusive of mechanical and electrical items, steel and platework, concrete, piping, earthworks and plant buildings
- $295 M for indirect costs inclusive of engineering charges, spares and first fills, freight charges, support and construction costs, commissioning and contingency/escalation (28%), plus taxes

In relation to the smelter upgrade cost listed in Table 1-7, the figure of $115 M covers direct costs inclusive of:

- $16.3 M for acid plant 5
- $26 M for the oxygen plant
- $7.4 M for the matte granulator
- $6.2 M for wet electrostatic precipitators (WESP)
- $53 M for smelter plant-wide services
- $6.3 M for miscellaneous facilities

Table 1-8 summarises the sustaining capital cost provisions in the Mineral Reserve cashflow model. These provisions account for:

Mining

- mining fleet replacements and component change-out
- ancillary fleet replacements

Processing

- longer term enhancements to the process plant and smelter
- periodic Isasmelt shutdowns

Table 1-8 Kansanshi sustaining cost provisions

Year	Mining ($M)	Processing ($M)	Smelting ($M)	Infrastructure and other ($M)	Total ($M)
2024	$77.0	$2.9	$9.1	$66.3	**$155.3**
2025	$151.4	$2.2	$25.0	$22.1	**$200.8**
2026	$168.7	$0.9	$1.8	$17.8	**$189.2**
2027	$141.7	$1.2	$2.5	$16.0	**$161.4**
2028	$96.8	$1.1	$0.9	$16.0	**$114.7**
2029	$42.3	$16.1	$32.3	$2.1	**$92.7**
2030	$42.2	$16.1	$2.3	$0.1	**$60.7**
2031	$42.7	$16.1	$2.3	$0.1	**$61.1**
2032	$42.7	$16.8	$2.3	$0.5	**$62.3**
2033	$42.6	$16.8	$32.3	$0.0	**$91.7**
2034	$137.4	$16.8	$2.3	$2.1	**$158.6**
2035	$137.4	$16.8	$2.3	$0.1	**$156.6**
2036	$137.4	$16.8	$2.3	$0.1	**$156.6**
2037	$136.8	$16.8	$32.3	$0.5	**$186.4**
2038	$134.8	$16.7	$2.3	$0.0	**$153.8**
2039	$32.0	$16.7	$2.3	$2.1	**$53.0**
2040	$37.7	$8.2	$2.3	$0.1	**$48.4**
2041	$26.1	$8.2	$32.3	$0.1	**$66.7**
2042	$24.3	$8.2	$2.3	$0.5	**$35.3**
2043	$18.7	$8.2	$2.3	$0.0	**$29.2**
2044	$12.7	$8.2	$2.3	$0.1	**$23.4**
2045	$5.7	$8.2	$32.3	$0.0	**$46.2**
2046	$5.0	$8.2	$2.3	$0.0	**$15.5**
2047					
2048					
2049					
2050					
2051					
TOTAL	**$1,694.2**	**$248.3**	**$230.7**	**$146.4**	**$2,319.6**

Infrastructure and other

Table 1-9 lists the closure cost provisions, as updated from the 2020 estimation. In the Mineral Reserve cashflow model this provision is spread between 2040 and 2050, i.e. into the years preceding and after scheduled final ore processing.

Table 1-9 Kansanshi closure cost provisions

	($M)
Closure components	
Dismantling of plant, buildings and related structures	$26.5
Demolition of steel buildings and structures	$0.0
Demolition of reinforced concrete structures buildings	$0.1
Rehabilitation of roads and paved surfaces	$2.2
Demolition of offices, workshops and residential buildings	$9.9
Demolition and rehabilitation of railway lines	$0.0
Fencing	$0.1
Disposal of other linear infrastructure	$1.7
Disposal and handling of waste	$4.9
Making good of infrastructure	$0.0
Subtotal	**$45.5**
Mining areas	
Sealing of shafts, adits and inclines	$0.4
Open pit rehabilitation including final voids and ramps	$5.6
Rehabilitation of overburden, processing residues, spoils and waste rock dumps	$1.7
Rehabilitation of processing waste deposits and evaporation ponds (polluting potential)	$27.3
Rehabilitation of subsided areas	$0.0
Subtotal	**$35.1**
Surface rehabilitation	
General surface rehabilitation	$8.4
Subtotal	**$8.4**
Surface runoff measures	
Dambo wetlands - reinstatement of aquatic health	$1.8
Subtotal	**$1.8**
Pre-site relinquishment aspects	
Initial monitoring and aftercare	$21.4
Subtotal	**$21.4**
P&Gs, contingencies and additional allowances	
Preliminary and general	$13.6
Contingencies	$0.0
Closure related aspects	$2.5
Subtotal	**$16.1**
Residual and latent aspects	
Residual aspects	$0.0
Latent aspects	$0.0
Contingencies on residual and latent aspects	$0.0
Subtotal	**$0.0**
Total scheduled closure costs	**$128.2**

1.6.5 Operating and metal costs

The currently estimated unit costs, profiled over the life of mine, are listed in Table 1-10.

The current estimates reflect comprehensive budgeting and projections which seek to rationalise the Project operating costs. Where possible, and particularly in the case of future mining costs, improved operating efficiencies have been considered in terms of the mining fleet and performance, and modifications to the ore and waste haulage profiles to cater for expanded trolley-assist infrastructure.

The estimated S3 plant operating costs have been derived from first principles, cross-referencing actual costs from Sentinel. The cost estimates account for power consumption, labour costs, reagents and other consumables. Work continues on developing a detailed operating budget for the S3 plant, focussing on further optimisation, identifying improvement efficiencies, and refining the operating costs going forward.

Table 1-10 Kansanshi operating cost estimates

Year	Mining costs			Mining costs	Processing Costs				Gold	G&A
	Total Ore	Reclaim	Waste		Oxide	Mixed	S2	S3	plant	costs
	$/bcm	$/bcm	$/bcm	$/bcm	$/t process	$/t process	$/t process	$/t process	$/oz	$/t process
2024	$7.67	$5.96	$7.67	$7.67	$13.63	$10.22	$9.97		$166.75	$1.80
2025	$6.39	$4.50	$6.39	$6.39	$10.35	$7.51	$7.94	$6.34	$114.15	$1.62
2026	$6.11	$4.43	$6.11	$6.11	$9.85	$7.30	$7.13	$7.34	$91.19	$1.41
2027	$5.89	$4.27	$5.89	$5.89	$9.96	$6.09	$9.11	$6.95	$90.42	$1.42
2028	$5.78	$4.08	$5.78	$5.78	$9.97	$7.18	$7.00	$7.03	$91.07	$1.43
2029	$5.66	$4.14	$5.66	$5.66	$10.05	$7.18	$7.00	$7.03	$98.78	$1.42
2030	$5.81	$4.33	$5.81	$5.81	$10.00	$7.18	$7.00	$7.03	$96.45	$1.42
2031	$5.80	$4.33	$5.80	$5.80	$10.04	$7.18	$7.00	$7.03	$98.85	$1.42
2032	$5.71	$4.33	$5.71	$5.71	$9.91	$7.13	$6.94	$6.95	$94.13	$1.36
2033	$5.85	$4.39	$5.85	$5.85	$9.92	$7.13	$6.95	$6.95	$93.33	$1.37
2034	$5.93	$4.36	$5.93	$5.93	$9.85	$7.13	$6.95	$6.95	$100.87	$1.37
2035	$5.86	$4.34	$5.86	$5.86	$9.84	$7.13	$6.95	$6.95	$99.78	$1.37
2036	$5.58	$4.28	$5.58	$5.58	$9.88	$7.13	$6.94	$6.95	$102.94	$1.36
2037	$5.63	$4.21	$5.63	$5.63	$9.85	$7.13	$6.95	$6.95	$99.19	$1.37
2038	$5.83	$4.33	$5.83	$5.83	$9.93	$6.94	$7.14	$6.96	$92.07	$1.37
2039	$6.34	$4.67	$6.34	$6.34	$10.00	$6.94	$7.14	$6.96	$101.14	$1.37
2040	$5.83	$4.48	$5.83	$5.83				$7.54	$117.76	$2.10
2041	$6.39	$4.74	$6.39	$6.39				$7.20	$126.70	$1.99
2042	$6.70	$5.01	$6.70	$6.70				$7.20	$119.10	$1.87
2043	$7.67	$5.84	$7.67	$7.67				$7.20	$125.47	$1.87
2044	$8.30	$6.34	$8.30	$8.30				$7.20	$138.46	$1.87
2045	$9.06	$6.96	$9.06	$9.06				$7.20	$139.72	$1.87
2046	$9.69	$7.39	$9.69	$9.69				$7.20	$129.17	$1.87
2047		$6.48						$7.20	$170.76	$1.87
2048		$6.44						$7.20	$170.76	$1.87
2049		$8.01						$7.15	$117.22	$2.73
2050										
Average	**$6.13**	**$5.30**	**$6.06**	**$6.08**	**$10.19**	**$7.22**	**$7.35**	**$7.08**	**$107.87**	**$1.55**

There is an additional annual average cost of approximately $0.30/t processed which could be assigned to corporate overheads.

The metal costs (treatment charges, refining charges (TCRCs)) were also reviewed and updated at different times, for optimisation and cashflow modelling, respectively, yielding summary figures as listed in Table 1-11.

Table 1-11 Kansanshi unit metal costs, excluding royalties

Transport, smelting and refining charges	Units	$/unit
Cathode SXEW costs		
Electrowinning	$/t cathode	395.86
	$/lb Cu	0.18
Cathode freight	$/t cathode	263.40
	$/lb Cu	0.12
Cathode insurance	$/t cathode	1.86
	$/lb Cu	0.00
Total cathode cost	**$/t cathode**	**661.12**
	$/lb Cu	**0.30**
Smelting/refining costs		
Concentrate grade	%	24.3
Concentrate road haualge to KCS	$/t con.	0.00
	$/lb Cu	0.00
KCS smelting cost	$/t con.	89.57
	$/lb Cu	0.17
Anode transport cost	$/t metal	218.58
	$/lb Cu	0.10
Insurance	$/t metal	1.84
	$/lb Cu	0.00
Refining charge	$/t metal	123.48
	$/lb Cu	0.06
Total smelting/refining cost	**$/lb Cu**	**0.32**
Gold refining charge	**$/lb oz**	**5.00**

1.6.6 Economic analysis

An economic analysis in the form of a cashflow model to support the Mineral Reserve estimate is listed in Table 1-12. This is an indicative cashflow model, inclusive of updated operating costs and metal charges (TCRC and royalties), presented pre-tax and post-tax.

Inputs and assumptions in this model are as follows:

1. The annual mining and processing production schedules are the same as those listed in Table 1-5 and Table 1-6, respectively.
2. The process recoveries for copper vary for each processing circuit according to equations developed from metallurgical performance analysis and projections.
3. For Mineral Reserve reporting transparency, the cashflow shows the insitu and recovered gold that relates directly to the gold grades carried from the Mineral Resource model.
4. The cashflow reflecting the inclusion of gravity gold is reported separately, for information only.
5. The annual gross revenues are calculated from the same metal prices as used in the pit optimisation process, i.e. \$3.50/lb Cu (or \$7,717/t Cu) and \$1,805/oz Au in the optimisation
6. The payability rates are 100% of cathode and 99.7% of anode, plus 96.3% of gold in concentrate.
7. No copper concentrate from Kansanshi is sold to external smelters and no concentrate is processed through the HPL circuit.
8. The Project capital costs and expenditure timeframes are as listed in Table 1-7.
9. The Project sustaining costs are as listed in Table 1-8.
10. The Project closure costs are as listed in Table 1-9.
11. The Project unit operating costs for mining, processing (i.e., through each circuit) are listed in Table 1-10.
    - The S3 unit processing costs are supported by a first principles basis estimate for reagents and power, and by comparison with S2 and Sentinel operating costs, whereas the unit processing costs for the other circuits, and for G&A (general and administration charges) are derived from current operating cost projections
    - corporate overhead costs are excluded
12. The unit metal costs in Table 1-11 (also referred to as TCRCs, or treatment and refining charges) are derived from current operating cost projections.
13. Gross royalties are estimated based on an average rate of 6.0% for copper, applicable from the government sliding scale for an adopted metal price of \$3.50/lb, and 6% for gold. A ZCCM royalty payment rate of 3.1% is also applicable.
14. The modelled taxes and royalty payments are net of VAT (value added tax) refunds owed to the Company in accordance with a VAT agreement with the Zambian government.
15. The applicable corporate tax rate is 30%.

The indicative, pre-tax undiscounted cashflow for the Mineral Reserve production schedule is \$8.2 B as from 2024, with an NPV reflecting a 10% discount rate equal to \$2.9 B (Table 1-12). On a post-tax basis, the undiscounted cashflow is \$6.3 B as from 2024, with an NPV equal to \$2.2 B. The corresponding internal rate of return (IRR) from 2024 is 42%.

When the revenue and costs associated with the additional gold are included (i.e., outside of the Mineral Reserve inventory), the pre-tax undiscounted cashflow and NPV figures rise to \$9.6 B and \$3.5 B, respectively. Post-tax, these figures are \$7.3 B and \$2.7 B, respectively. The corresponding IRR from 2024 is 50%.

Table 1-12 Kansanshi Mineral Reserve cashflow model summary

PHYSICALS	UNITS	FROM 2024	2024 - 2028	2029 - 2033	2034 - 2038	2039 - 2043	2044 - 2048	2049 - 2050
MINING								
Total ore	**Mbcm**	**351.6**	86.7	104.1	96.5	48.7	15.5	
Total waste	**Mbcm**	**1,069.3**	253.2	294.6	298.0	204.7	18.7	
Total mined	**Mbcm**	**1,420.8**	**339.9**	**398.8**	**394.5**	**253.4**	**34.3**	
Total ore	**Mt**	**935.2**	213.3	279.1	264.5	134.7	43.6	
Total waste	**Mt**	**2,718.3**	573.3	764.4	779.7	548.2	52.7	
Total mined	**Mt**	**3,653.6**	**786.7**	**1,043.5**	**1,044.2**	**682.9**	**96.3**	
ore density	t/bcm	**2.66**	2.46	2.68	2.74	2.77	2.80	
waste density	t/bcm	**2.54**	2.26	2.59	2.62	2.68	2.81	
total density	t/bcm	**2.57**	2.31	2.62	2.65	2.69	2.81	
Strip ratio		**2.9**	2.7	2.7	2.9	4.1	1.2	
TOTAL FEED TO PLANT								
Total plant feed	**Mt**	**1,104.8**	**231.8**	**272.8**	**279.7**	**165.3**	**137.0**	**18.1**
	%TCu	**0.52**	**0.57**	**0.59**	**0.54**	**0.49**	**0.34**	**0.33**
	g/tAu	**0.11**	**0.12**	**0.11**	**0.10**	**0.10**	**0.08**	**0.07**
	kt	**5,771.6**	**1,313.6**	**1,609.8**	**1,509.6**	**813.5**	**464.9**	**60.2**
	k(t)oz	**3,768.0**	**915.5**	**973.0**	**922.4**	**556.6**	**362.0**	**38.6**
AVERAGE RECOVERIES								
Oxide leach	Cu	**40.4%**	41.7%	40.6%	37.2%	45.2%		
Oxide flotation	Cu	**32.7%**	32.3%	32.8%	34.0%	29.8%		
	Au	**24.0%**	24.0%	24.0%	24.0%	24.0%		
Mixed flotation	%	**67.7%**	69.3%	68.3%	66.4%	60.3%		
	%	**32.0%**	32.0%	32.0%	32.0%	32.0%		
Sulphide S2 & S3 flotation	%	**87.8%**	89.6%	89.4%	90.2%	88.7%	76.4%	68.4%
	%	**37.5%**	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%
Overall average	**%**	**82.8%**	**82.0%**	**83.1%**	**83.8%**	**86.0%**	**76.4%**	**68.4%**
	%	**35.1%**	**33.8%**	**34.7%**	**35.1%**	**36.5%**	**37.5%**	**37.5%**
METAL RECOVERED								
Oxide leach	kt	**284.2**	91.8	101.5	67.8	23.1		
Oxide flotation	kt	**230.3**	71.2	81.9	62.0	15.2		
	k(t)oz	**104.3**	38.9	32.6	26.4	6.4		
Mixed flotation	kt	**625.2**	222.4	197.8	174.0	31.0		
	k(t)oz	**177.3**	70.9	51.8	43.6	11.0		
Sulphide S2 & S3 flotation	kt	**3,636.6**	691.9	957.1	960.5	630.5	355.3	41.2
	k(t)oz	**1,042.2**	199.6	253.1	253.5	185.8	135.7	14.5
Total metal recovered	**kt**	**4,776.3**	**1,077.4**	**1,338.2**	**1,264.3**	**699.9**	**355.3**	**41.2**
	k(t)oz	**1,323.8**	**309.3**	**337.6**	**323.5**	**203.2**	**135.7**	**14.5**

CASHFLOW SUMMARY	UNITS	FROM 2024	2024 - 2028	2029 - 2033	2034 - 2038	2039 - 2043	2044 - 2048	2049 - 2050
PAYABLE COPPER								
Total copper metal produced								
copper in cathode	kt	**284.2**	91.8	101.5	67.8	23.1	0.0	0.0
copper in anode	kt	**4,375.3**	959.9	1,204.6	1,165.4	659.2	346.1	40.1
Total Cu metal produced	**kt**	**4,659.5**	**1,051.8**	**1,306.1**	**1,233.2**	**682.3**	**346.1**	**40.1**
Total payable metal								
copper in cathode	kt	**284.2**	91.8	101.5	67.8	23.1	0.0	0.0
copper in anode	kt	**4,362.1**	957.1	1,201.0	1,161.9	657.2	345.0	40.0
Total Cu metal sold	**kt**	**4,646.4**	**1,048.9**	**1,302.4**	**1,229.7**	**680.3**	**345.0**	**40.0**
PAYABLE GOLD								
Gold in concentrate								
from oxide circuit	k(t)oz	**104.3**	38.9	32.6	26.4	6.4	0.0	0.0
from mixed circuit	k(t)oz	**177.3**	70.9	51.8	43.6	11.0	0.0	0.0
from sulphide circuit	k(t)oz	**1,042.2**	199.6	253.1	253.5	185.8	135.7	14.5
Total gold in Concentrate	**k(t)oz**	**1,323.8**	309.3	337.6	323.5	203.2	135.7	14.5
Total gold produced	**k(t)oz**	**1,323.8**	**309.3**	**337.6**	**323.5**	**203.2**	**135.7**	**14.5**
Total payable gold								
gold in concentrate	k(t)oz	**1,295.2**	302.6	330.3	316.6	198.8	132.8	14.1
gold in anode	k(t)oz	**1,173.3**	278.2	300.1	286.2	178.9	117.7	12.3
Total Au metal sold	**k(t)oz**	**1,173.3**	**278.2**	**300.1**	**286.2**	**178.9**	**117.7**	**12.3**
GROSS REVENUE								
Copper revenue	**$M**	**$35,852.1**	**$8,093.3**	**$10,049.9**	**$9,488.8**	**$5,249.2**	**$2,662.3**	**$308.6**
Gold revenue	**$M**	**$2,117.8**	**$502.1**	**$541.6**	**$516.5**	**$322.8**	**$212.5**	**$22.2**
Total revenue	**$M**	**$37,969.8**	**$8,595.4**	**$10,591.5**	**$10,005.3**	**$5,572.0**	**$2,874.8**	**$330.7**
CAPITAL COSTS								
S3 mining fleet	$M	**$178.3**	$178.3					
S3 expansion	$M	**$623.9**	$623.9					
SE Dome pre-strip	$M	**$236.0**	$236.0					
Smelter expansion and upgrades	$M	**$75.0**	$75.0					
Process plant upgrade	$M	**$13.2**	$13.2					
TSF expansion	$M	**$116.0**	$116.0					
Other site infrastructure and ancillary equipment	$M	**$27.8**	$27.8					
TSF 3	$M	**$94.0**	$25.0	$69.0				
Other stripping	$M	**$182.9**	$182.9					
Dewatering Shaft & Associated	$M	**$21.6**	$21.6					
New Trolley Line Installation	$M	**$30.7**	$30.7					
Second primary crusher	$M	**$70.0**	$70.0					
Gravity gold	$M	**$0.3**	$0.3					
Other (aggregate crusher, HPL oxygen upgrade, and other)	$M	**$18.5**	$18.5					
Closure costs	$M	**$128.2**				$53.6	$53.3	$21.3
Total capital costs	**$M**	**1,816.2**	**$1,619.0**	**$69.0**	**$0.0**	**$53.6**	**$53.3**	**$21.3**
SUSTAINING COSTS								
Mining sustaining	$M	**$1,694.2**	$635.7	$212.4	$683.9	$138.8	$23.4	$0.0
Process sustaining	$M	**$248.3**	$8.3	$81.9	$83.9	$49.6	$24.7	$0.0
Smelter sustaining	$M	**$230.7**	$39.3	$71.5	$41.5	$41.5	$36.9	$0.0
Other sustaining	$M	**$146.4**	$138.1	$2.7	$2.7	$2.7	$0.1	$0.0
Total sustaining costs	**$M**	**$2,319.6**	**$821.4**	**$368.5**	**$812.0**	**$232.6**	**$85.1**	**$0.0**
OPERATING COSTS								
Mining								
Ore mined to direct feed and to s/pile	$M	**$2,157.0**	$541.8	$600.6	$557.4	$319.8	$137.4	$0.0
Ore reclaimed from stockpile	$M	**$796.4**	$221.2	$70.2	$85.6	$78.6	$280.2	$60.6
Waste mined to dump	$M	**$6,478.3**	$1,588.8	$1,698.9	$1,717.3	$1,312.5	$160.8	$0.0
Stockpile relocation	$M	**$64.8**	$64.8	$0.0	$0.0	$0.0	$0.0	$0.0
Deduct SED pre-strip and stripping	$M	**-$418.9**	-$418.9	$0.0	$0.0	$0.0	$0.0	$0.0
Subtotal mining costs	**$M**	**$9,077.6**	**$1,997.7**	**$2,369.7**	**$2,360.3**	**$1,710.8**	**$578.4**	**$60.6**
Processing and G&A								
Oxide feed	$M	**$1,190.1**	$392.0	$364.6	$360.5	$73.0	$0.0	$0.0
Mixed feed	$M	**$1,020.5**	$326.7	$278.6	$318.6	$96.6	$0.0	$0.0
S2 feed	$M	**$1,462.1**	$513.2	$471.6	$426.5	$50.9	$0.0	$0.0
S3 feed	$M	**$4,586.4**	$621.4	$908.2	$952.5	$988.8	$986.3	$129.1
G&A	$M	**$1,708.5**	$348.9	$381.0	$382.3	$291.0	$256.0	$49.3
Subtotal processing costs	**$M**	**$9,967.6**	$2,202.1	$2,404.0	$2,440.4	$1,500.3	$1,242.3	$178.5
Total operating costs	**$M**	**$19,045.2**	**$4,199.8**	**$4,773.7**	**$4,800.7**	**$3,211.1**	**$1,820.7**	**$239.1**
METAL COSTS (TCRCs AND ROYALTIES)								
Cathode costs	$M	**$175.7**	$57.6	$61.2	$43.1	$13.7	$0.0	$0.0
KCS smelting/refining costs	$M	**$3,219.9**	$721.0	$894.1	$857.5	$472.7	$245.0	$29.7
Gold refining costs	$M	**$5.9**	$1.4	$1.5	$1.4	$0.9	$0.6	$0.1
Copper royalty (net)	$M	**$1,908.9**	$398.7	$486.2	$526.9	$317.4	$161.0	$18.7
Gold royalty (net)	$M	**$112.6**	$24.6	$26.0	$28.6	$19.4	$12.8	$1.3
ZCCM royalty	$M	**$1,177.1**	$266.5	$328.3	$310.2	$172.7	$89.1	$10.3
Total metal costs	**$M**	**$6,600.0**	**$1,469.8**	**$1,797.3**	**$1,767.7**	**$996.8**	**$508.5**	**$60.0**
MINERAL RESERVE CASHFLOW; PRE-TAX								
Undiscounted (from 2024)	$M	**$8,188.8**	**$485.4**	**$3,583.0**	**$2,624.9**	**$1,077.9**	**$407.2**	**$10.4**
NPV_{10} (from 2024)	$M	**$2,851.9**						
IRR (from 2024)	%	**46%**						
Payback year (from 2024)	**Year**	**2028**						
MINERAL RESERVE CASHFLOW; POST-TAX								
Undiscounted (from 2024)	$M	**$6,272.5**	**$416.5**	**$2,953.6**	**$1,859.1**	**$780.6**	**$273.9**	**-$11.3**
NPV_{10} (from 2024)	$M	**$2,223.1**						
IRR (from 2024)	%	**42%**						
Payback year (from 2024)	**Year**	**2028**						

If an April 2024 consensus, long term copper price of $4.02/lb was adopted, the pre-tax undiscounted cashflow and NPV under these circumstances, would be $13.3 B and $5.2 B, respectively. Post-tax, the undiscounted cashflow and NPV would be $9.9 B and $4.0 B, respectively, and the IRR (from 2024) would be 89%.

At the higher copper price and with the inclusion of additional gold outside of the Mineral Reserve inventory, the pre-tax cashflow and NPV figures become $14.8 B and $5.8 B, respectively. Post-tax, the respective figures are $10.9 B and $4.4 B, with an IRR (from 2024) of 104%.

At the Mineral Reserve copper price of $3.50/lb Cu, under post-tax circumstances and starting from 2024, the payback year is 2028. At the higher copper price, the payback year would be 2026.

1.7 Conclusions and recommendations

1.7.1 Mineral Resource estimates

Drill hole data used in the Mineral Resource estimate for this Technical Report covers the extents of the Kansanshi deposits. Continued improvements in sample QAQC demonstrates representative assay results for both diamond drilled and reverse circulation drilled samples. Improved geological understanding and resulting models conform with the exposed deposit geology within the current pits. The large number of close spaced reverse circulation drill holes adds confidence to the interpretation of local geology and to the grade continuity for the respective mineralisation domains. The Qualified Person, Carmelo Gomez Dominguez of FQM, believes that this Mineral Resource estimate is representative of the prevailing geology and drilled sample data.

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

1. Geology modelling methods for improving constraints on mineralisation should continue to be tested and developed.
2. Infill drilling for improving on accuracy of estimates should continue to target areas having wider grid spacings.
3. Development of data acquisition techniques for improving support in defining vein volumes should continue.
4. Reconciliation information continues to assist in understanding estimation variances.

1.7.2 Mineral Reserve estimates

Relative to the Mineral Reserve estimate that was produced for the 2020 Technical Report (FQM, September 2020), and reflective of mining depletion to the end of 2023 (AIF, March 2024), the estimate completed for this Technical Report indicates an overall uplift of 31% and 19% respectively, for Mineral Reserve ore tonnes and insitu copper metal, respectively. This uplift is largely attributable to Mineral Resource model updates, specifically related to the inclusion of new drilling/assaying data, estimation changes and improvements.

The Qualified Person, Michael Lawlor of FQM, believes that the Kansanshi Mineral Reserve estimate reflects an achievable mining plan to support the proposed S3 expansion, and with production sequencing taking into account phased mining progression and increased material movement profiles (and hence equipment usage). There is considered to be minimal risk attributable to the conventional mining methods and the primary equipment proposed for the scale of expanded mining operations.

That said, certain operational improvements will be required to ensure that the projected annual material movements are realised as the mine transitions towards a bulk rather than selective mining operation, also facing higher waste stripping ratios. As an aside to these operational improvements, future mine planning should continue working towards the assurance of reduced unit mining costs that account for an expanded trolley-assist haulage network and the modelling of discrete ore and waste haulage profiles.

One particular aspect that has been extensively analysed during mine planning for this Technical Report is that of planned and unplanned metal loss projections. Whilst a review of current trends has been given consideration, a numerical method has been enhanced to estimate and quantify metal loss trends going forward, especially when catering for diverse styles of mineralisation and the increased mining production rates. Whilst the methodology adopted in this instance is believed to be more comprehensive than that adopted for the Mineral Reserve estimated reported in the 2020 Technical Report, it is recommended that continuous improvement should remain a focus point.

1.7.3 Processing

The processing facilities at Kansanshi provide flexibility in the treatment of oxide, mixed and sulphide ores through various circuit options.

Process enhancements continue to be made; the three major initiatives being:

1. Additional sulphide cleaner capacity was installed in 2021 to improve concentrate grades and to assist in increasing concentrate throughput at the smelter.
2. The cleaner circuit is to be upgraded further for the S3 expansion.
3. Several modifications to the smelter and acid plants to increase concentrate treatment up to 1.6 Mtpa from the nameplate 1.2 Mtpa.
4. Completion of the S3 Project in 2025 to increase the sulphide treatment rate by 27.5 Mtpa.

The Qualified Person, Andrew Briggs of FQM, is satisfied that the plans in place for these expansions will provide the stated benefits.

ITEM 2 INTRODUCTION

2.1 Purpose of this Technical Report

This Technical Report on the Kansanshi Operations (the property) has been prepared by Qualified Persons (QPs) Carmelo Gomez Dominguez, Michael Lawlor and Andrew Briggs of First Quantum Minerals Pty Ltd (FQM, the issuer).

The purpose of this Technical Report is to document updated Mineral Resource and Mineral Reserve estimates for the property, and to provide a commentary on the status of the operations and proposed S3 expansion.

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 Securities Administrators' National Instrument 43-101 *Standards of Disclosure for Mineral Properties* (NI 43-101 or the Instrument).

The effective date for the Mineral Resource estimate and for the Mineral Reserve estimate is 31st December 2023.

2.3 Qualified Persons and authors

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

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

2.4 Principal sources of information

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

2.5 Site visits

The Qualified Persons (QPs) have visited the site, as follows:

- Carmelo Gomez last visited the Kansanshi Operations in June 2022. Mr Gomez inspected drill core and drilling sites, reviewed geological, data collection and sample preparation procedures, and carried out independent data verification.
- Michael Lawlor last visited the Kansanshi Operations in March 2023. Mr Lawlor visited the operating mine areas and the S3 expansion site.
- Andrew Briggs last visited the Kansanshi Operations in July 2023. Mr Briggs visited all accessible areas of the site, particularly the processing plant, the tailings storage facilities and the smelter.

Table 2-1 QP details

Name	Position	NI 43-101 Contribution
Carmelo Gomez Dominguez	**Group Principal Geologist, Mine and Resources**	**Author and Qualified Person**
BSc Hons (Geology), MAusIMM, EurGeol	FQM (Australia) Pty Ltd	Items 1, 7 to 12, 14, 25 and 26
Michael Lawlor	**Mine Technical Advisor**	**Author and Qualified Person**
BEng Hons (Mining), MEngSc, FAusIMM	FQM (Australia) Pty Ltd	Items 1 to 6, 15, 16 and 18 to 26
Andrew Briggs	**Group Consulting Metallurgist**	**Author and Qualified Person**
BSc (Eng), ARSM, FSAIMM	FQM (Australia) Pty Ltd	Items 13, 17 and 21 (in respect of processing and G&A costs only), 25 and 26

2.6 Conventions and definitions

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

The conventional chemical abbreviation for copper of Cu is used throughout this report, whilst the abbreviation for gold is Au. ASCu is used to denote Acid Soluble Copper and TCu is used to denote Total Copper. TCRC is an abbreviation for (copper concentrate) treatment charges and refining charges.

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

Table 2-2 Terms and definitions

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

Dollar references followed by “M” mean millions of dollars, and followed by “B” mean billions of dollars. A billion dollars equals 1,000 million dollars. P+P refers to Proven and Probable Mineral Reserves combined.

Throughout this Technical Report, reference is made alternately to the South East deposit and to the South East Dome deposit. The deposit names are synonymous.

ITEM 3 RELIANCE ON OTHER EXPERTS

The authors of this Technical Report do not disclaim any responsibility for the content contained herein, with the exception of certain information included in the Item 22 Economic Analysis. This information, provided by the Company's internal taxation advisors, relates to the applicable corporate tax rate in Zambia, the estimated Project taxable income and the tax to be paid. The modelled taxes, and royalty payments, are net of expected VAT (value added tax) refunds.

The authors of this Technical Report have relied on this information for the purposes of the Project economic analysis in Item 22.

ITEM 4 PROPERTY DESCRIPTION AND LOCATION

4.1 Project description

Kansanshi is one of the larger copper mining and processing operations in the world, with an annual average production of about 245,000 tonnes of recovered metal over the last ten years. The mine is the largest copper mine in Africa, and in respect of the Mineral Reserve processing schedule, has an estimated remaining life of 26 years from January 2024. As at December 2023, the operations employed a workforce of over 8,000 people both directly and as contractors. An additional project workforce of approximately 700 direct employees and contractors were also employed as at this date.

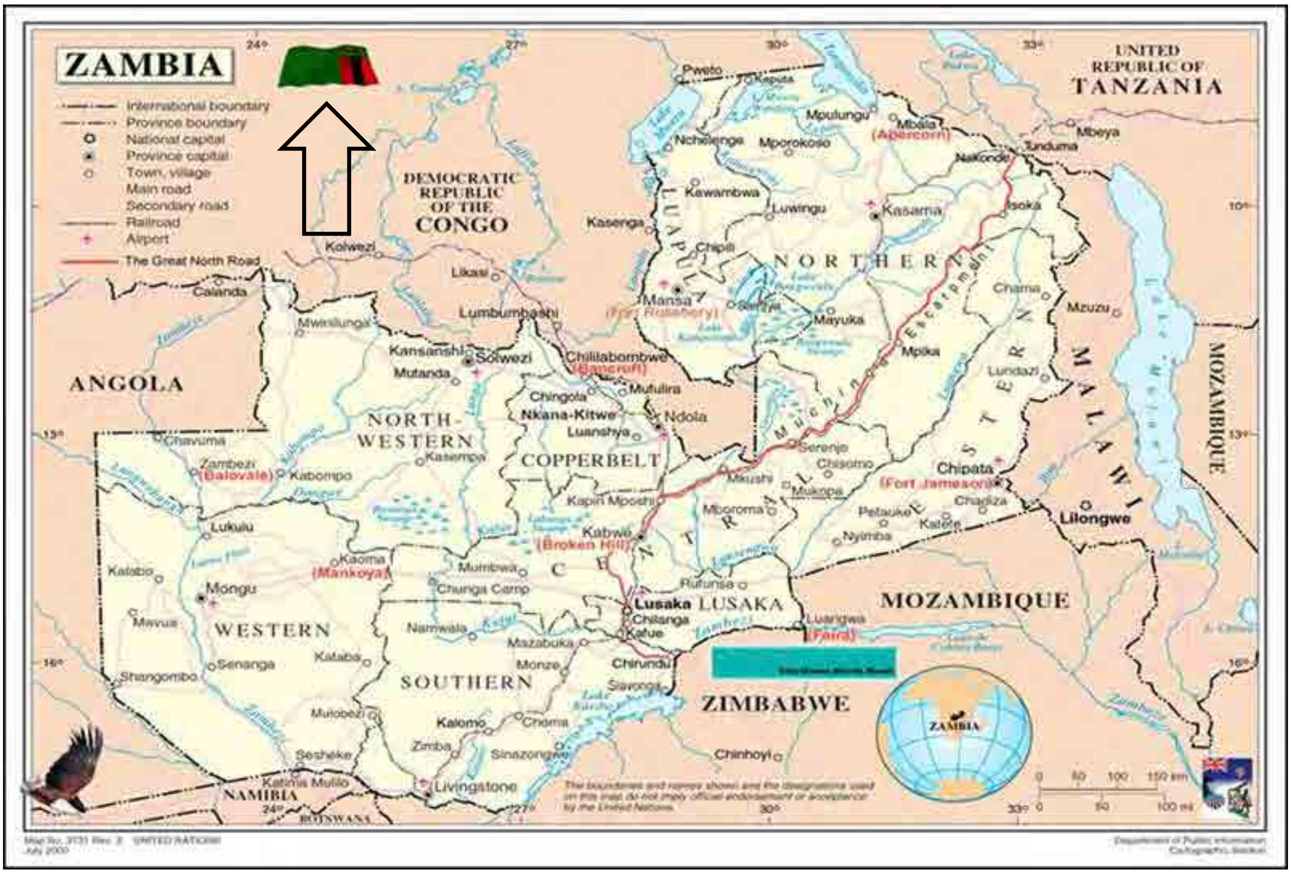
Conventional open pit mining has been carried out to date in two adjoining pits (Main and North West), with a third (South East Dome) commencing in 2024. The current facilities are able to process multiple copper mineralisation styles and produce cathode copper, in addition to anode from an on-site smelter. Gold production is also a significant revenue stream.

The increased sulphide ore processing capacity attributable to the S3 expansion, and supported by mining expansion into the South East Dome deposit, are the subject of this Technical Report, with these developments significantly increasing the average annual copper production above 250 kt of recovered metal between 2030 and 2035.

4.2 Project location

The Kansanshi Operations are located in the North-Western Province of Zambia; approximately 10 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: United Nations, July 2000)



4.3 Tenure and property area

FQM acquired its 80% interest in the Kansanshi Operations in 2001 from Cyprus Amax Minerals Corporation. The remaining 20% is owned by the parastatal company Zambia Consolidated Copper Mines - Investment Holdings (ZCCM - IH). The site operating entity is Kansanshi Mining PLC (KMP).

KMP is the owner of the 7057-HQ-LML mining licence, which was issued on the 7th March 1997 as licence No. LML 16 and was valid until the 7th March 2022. In February 2022 the mining licence was renewed for a period of 25 years and is now valid until the 6th March 2047. Conditions of grant of the renewed licence remain unchanged. The mining licence allows KMP to explore and mine copper, cobalt, gold, silver, tellurium, selenium and sulphur.

In 2009, 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 area of the licence was subsequently increased to 24,865 ha in November 2014 and remained unchanged following renewal of the mining licence in 2022. The original and current extents of the mining licence are shown in Figure 4-2. When altering the mining licence boundaries in 2014, the Company relinquished parts that had been compromised by encroachment from the Solwezi town. The licence boundaries were also extended to the east to provide for future expansion of the TSF2 tailings dam.

The coordinates for 7057-HQ-LML are listed in Table 4-1.

A mining right confers exclusive rights to develop and operate a mine within the boundary of the mining licence area. Surface rights are held by the Company under six Certificates of Title, the combined extents of which are shown in Figure 4-3.

Table 4-1 Extents of Kansanshi Mining Licence, 7057-HQ-LML (coordinates are ARC1950, SUTM35)

Beacon	Northing	Easting
1	8,670,350.920	429,971.928
2	8,670,372.976	440,131.718
3	8,667,055.763	440,138.365
4	8,667,060.742	442,677.980
5	8,670,377.945	442,671.622
6	8,670,392.771	451,017.093
7	8,667,259.820	451,022.223
8	8,667,262.159	452,473.410
9	8,662,102.200	452,481.640
10	8,662,103.640	453,388.568
11	8,658,970.678	453,393.483
12	8,658,956.062	444,869.844
13	8,656,007.501	444,875.328
14	8,655,997.973	439,979.262
15	8,657,287.938	439,976.640
16	8,657,281.539	436,893.820
17	8,657,465.860	436,893.424
18	8,657,457.101	432,903.751
19	8,663,170.202	432,890.846
20	8,663,170.612	433,072.263
21	8,666,856.406	433,063.983
22	8,666,849.328	429,980.133
23	8,670,350.920	429,971.928
24	8,670,350.920	429,971.928

Figure 4-2 Expanded Kansanshi Mining Licence extents, as at 31st December 2023 (coordinates are ARC1950, SUTM35) (source: FQM)

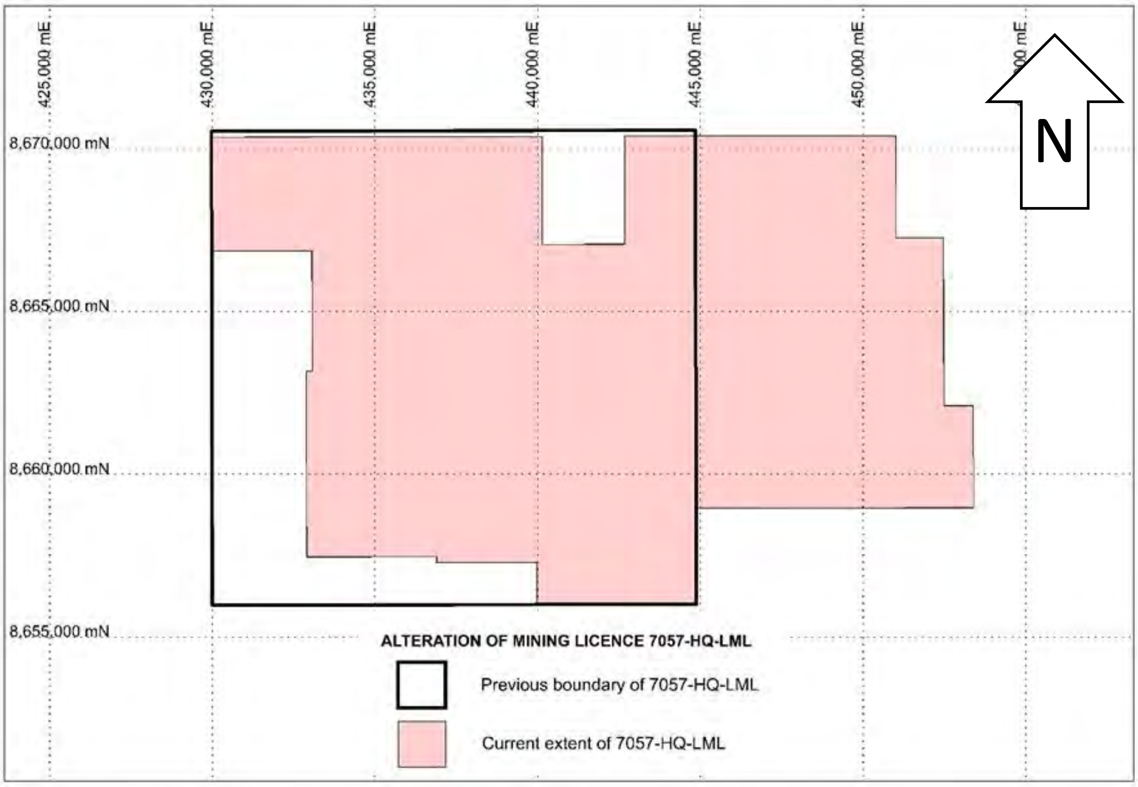


Figure 4-3 Kansanshi Mining Licence and Surface Rights extents as at 31st December 2023 (coordinates are ARC1950, SUTM35) (source: FQM)

4.4 Royalties, rights, payments and agreements

Changes to the mining royalty rate announced in September 2022 were implemented effective from 1st January 2023. The sliding scale mineral royalty rate on copper to be applied incrementally in each price range is currently as follows:

- 4% when LME average copper price for the month is below US$4,000/t Cu
- 6.5% when copper price = US$4,000 - <US5,000/t Cu
- 8.5% when copper price = US$5,001 - <US$7,000/t Cu
- 10% when copper price = US$7,001/t Cu, or higher

Deductibility of mineral royalty taxes was reintroduced effective 1st January 2022. Royalties on gold production have remained at 6%. The applicable corporate tax rate is 30%.

At the time of preparing this Technical Report, an export levy on gold had been suspended indefinitely and the import duty on copper and cobalt concentrate levied at 0% and 5%, respectively.

In December 2022, an agreement was entered into between the Company and ZCCM-IH to convert ZCCM-IH's dividend rights into a 3.1% revenue royalty. The Company now holds 100% of the dividend rights relating to the Kansanshi Operations.

Other than the joint-ownership with ZCCM - IH, the Kansanshi Operations are not subject to any known third-party royalties, rights, payments, agreements or encumbrances.

4.5 Environmental liabilities

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

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

The Kansanshi Mine closure plan is reviewed annually. At the end of December 2023, the asset retirement obligation (ARO) at Kansanshi was estimated to be $103.8 M for unscheduled closure and $128.2 M for scheduled closure. In accordance with national legislation, KMP contributes to an Environmental Protection Fund administered by the Zambian Mines Safety Department.

The Company is implementing an Environmental Management System (EMS) based on ISO 14001:2015 Standard. The Standard provides a structured approach to environmental management including pollution prevention, legal compliance and continued environmental improvement.

There are no known environmental risks that could materially affect the potential development of the Mineral Resources or exploitation of the Mineral Reserves as disclosed in this report.

4.6 Permits

Kansanshi has in place all applicable environmental and associated permits issued by the Zambian Environmental Management Agency (ZEMA) and other authorities. Site environmental permits are issued every three years; these set out requirements for the management of waste, water and air emissions, ozone depleting substances (ODS) and chemicals. Relevant legislation includes the Mines and Minerals Development Act, 2015, Water Resources Management Act, 2011 and Environmental Management Act, 2011.

4.7 Factors and risks which may affect access or title

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

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

5.1 Accessibility

The T5 national highway provides the primary road access to the Kansanshi Operations site. The highway is a sealed 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 Kansanshi Operations 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 mm 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 disruptive influence 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 masl (metres above sea level). 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 flows predominantly south towards the Mutanda River, part of the Kafue River system, eventually draining into the Zambezi River. Direct surface runoff 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. There is limited vegetation within the Operations site that is not already impacted upon by people and by agricultural activities.

5.5 Local resources

The nearest major population centre is at nearby Solwezi. 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 2023, the Kansanshi Operations employed over 8,700 persons directly and indirectly, in the operations and projects workforces. In relation to the S3 expansion, the peak month for personnel numbers is August 2024 with 1,950 individual contractors and employees on site.

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 the Company's presence in Solwezi, the infrastructure in the 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 Operations site, both of which were 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.

Over 200 houses in Solwezi have been provided for Company personnel and families.

Improved power supply to the Kansanshi site is supplied through the parastatal company Zambia Electricity Supply Corporation Ltd (ZESCO). Current KMP consumption of around 157 MVA is expected to increase to 230 MVA owing to the site expansion projects. Hydroelectric generation from the Kariba South Bank Dam, the Kafue Gorge Dam and from the Itezi Tezhi Dam, supplies power into the ZESCO grid. A single 330 kV line runs from the Copperbelt to the KMP site from where a ZESCO substation transforms the power down to 33 kV for mine usage.

5.7 Sufficiency of surface rights

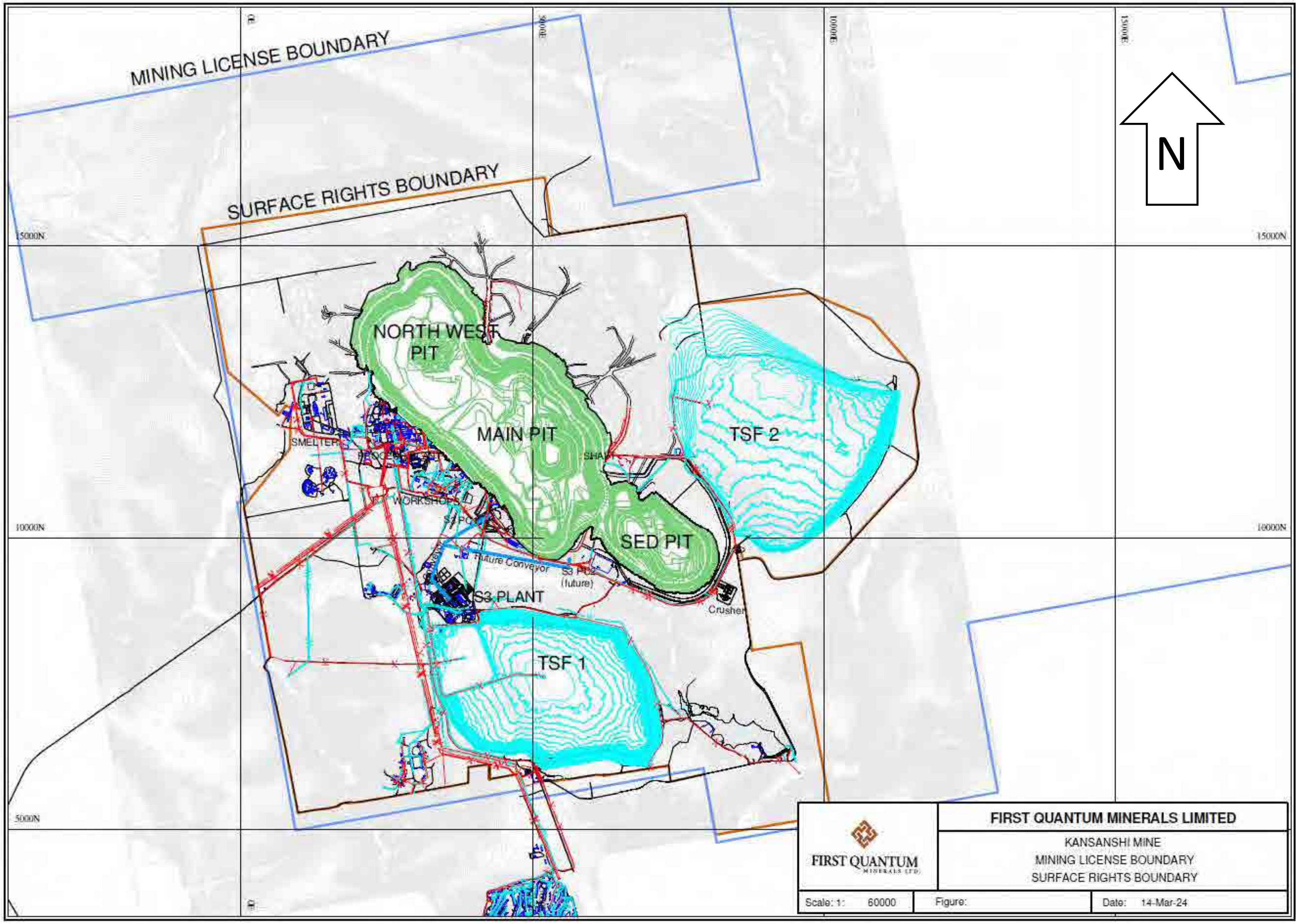
The 7057-HQ-LML mining licence boundaries were expanded in November 2014 (Item 4.3). The mining surface rights, within the mining licence boundaries, are held by the Company under six Certificates of Title. The Company can develop any part of the mining licence outside of the area for which surface rights are held, unless that part of the licence area is held under Title by another party.

Under these circumstances, the Company would have to purchase the land from the current owner, or otherwise enter into an Access Agreement with that owner. In the case of the latter, the owner cannot unreasonably refuse a request to develop the land. The Mines Act is superior to the Lands Act which specifically excludes land held under a mining right in the definition of land.

Figure 5-1 shows the extents of the proposed ultimate pits, and the extents of the existing waste dumps, stockpiles and tailings storage facilities, relative to the Mining Licence and surface rights boundaries. Expansion of TSF1 within the rights boundary is constrained to the north by the presence of a dambo, and to the south east by the presence of a water dam.

There is space for extension of the waste dump to the north and north west within the rights boundary, and there is also space for expansion of TSF2 within the rights boundary. In addition to expansion of the TSF2 footprint to cater for increased production, there is also the ability to raise one or both of the existing TSFs.

Figure 5-1 Kansanshi Mining Licence and surface rights boundaries – as at December 2023 (source: FQM)



ITEM 6 HISTORY

6.1 Prior ownership

Commercial open pit operations commenced at Kansanshi in 1977 when ZCCM began processing high grade copper oxide ore. Mining and processing continued intermittently until 1986 when operations ceased due to prevailing economic conditions at that time. Mining operations resumed around 1988 and 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 Operations, which, in turn were acquired by FQM in August 2001.

6.2 Exploration and development work

Cyprus Amax drilled 387 diamond core (DD) drill holes between 1997 and 1999 for 81,693 m of core. In addition, 34 reverse circulation (RC) holes for 5,147 m were drilled for exploration, condemnation and water monitoring purposes.

Since 2001, FQM undertook RC and DD in-fill programmes. From 2009, DD and RC drilling intensified, including a focus of drilling across the South East Dome region.

More recently, since 2019, DD drilling has focussed on targeting extension opportunities across the North West, Main and the South East Dome deposits. Some 215 DD holes for 69,619 m of drilled core were completed between 2019 and the start of 2024. In contrast, RC drilling has focussed on providing short term mining coverage for the purposes of ore control. RC ore control drilling is confined to within the open pit areas and uses a drill grid spacing of 12.5 m by 15 m.

A broader-scale brownfields campaign was launched in 2023 with emphasis on targeting new domal structures within the mine lease, adjacent to and along strike from the Kansanshi dome. In addition to reviewing previous mapping, magnetic, electromagnetic (EM) and magnetotelluric data, a new airborne EM survey was acquired across the licence to assist with targeting. This campaign remains underway with 11 diamond holes drilled in 2023 (5,243 m in total).

6.3 Previous Mineral Resource estimates

The previous Mineral Resource estimate was stated in the 2020 Technical Report. The reported statement complied with the reporting requirements of NI 43-101. The 2020 Technical Report estimate used the data available at the time to estimate copper, acid soluble copper and gold grades, using various estimation techniques, into an empty geology block model having vein and strata domains. The Mineral Resource estimate, inclusive of the Mineral Reserve estimate, includes South East Dome mineralisation. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The statement is reproduced in Table 6-1.

Table 6-1 Kansanshi Mineral Resource statement, excluding stockpiles, at a 0.2% Total Copper cut-off grade and depleted of mining as at 30th June 2020

Classification	Volume (millions)	Tonnes (millions)	Density (t/m^3)	Cu (%)	ASCu (%)	Au (g/t)
Measured	122.8	306.6	2.50	0.71	0.13	0.12
Indicated	232.4	633.0	2.72	0.65	0.07	0.12
Total Measured & Indicated	**355.2**	**939.6**	**2.65**	**0.67**	**0.09**	**0.12**
Inferred	60.5	166.5	2.75	0.58	0.04	0.11

6.4 Previous Mineral Reserve estimates

DumpSolver Pty Ltd (DumpSolver) produced the 2012 Technical Report Mineral Reserve estimate for Kansanshi (Journet and Cameron, DumpSolver, 2012). The estimate was subsequently updated for the 2015 Technical Report (FQM, May 2015) and again for the 2020 Technical Report.

The 2020 statement is reproduced by pit and classification in Table 6-2. The reported Mineral Reserve was based on an economic cut-off grade which accounted for longer-term copper metal and gold price projections of $3.00/lb ($6,614/t) and $1,200/oz, respectively. The Mineral Reserve inventory reflected the optimisation and phased pit designs that were current at the time, and a corresponding mining and processing production schedule.

Table 6-2 Kansanshi Mineral Reserve statement, depleted for mining as at 30th June 2020, and based on a $3.00/lb Cu price

MINERAL RESERVE AT 30th JUNE 2020 (AT $3.00/lb Cu and $1200/ounce Au)					
Oxide Ore					
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
MAIN & NW	Proven	51.4	0.87	0.47	0.13
	Probable	55.4	0.68	0.37	0.11
	Total P+P	**106.8**	**0.77**	**0.42**	**0.12**
SE DOME	Proven	2.7	0.55	0.26	0.06
	Probable	4.8	0.65	0.32	0.11
	Total P+P	**7.4**	**0.61**	**0.30**	**0.09**
TOTAL	Proven	54.1	0.86	0.46	0.13
	Probable	60.2	0.68	0.37	0.11
	Total P+P	**114.2**	**0.76**	**0.41**	**0.12**
Mixed Ore					
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
MAIN & NW	Proven	59.3	0.57	0.16	0.10
	Probable	55.6	0.56	0.15	0.09
	Total P+P	**114.9**	**0.56**	**0.16**	**0.09**
SE DOME	Proven	7.7	0.48	0.14	0.06
	Probable	16.3	0.57	0.16	0.08
	Total P+P	**24.0**	**0.54**	**0.15**	**0.07**
TOTAL	Proven	67.0	0.56	0.16	0.10
	Probable	71.9	0.56	0.15	0.08
	Total P+P	**138.9**	**0.56**	**0.16**	**0.09**
Sulphide Ore					
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
MAIN & NW	Proven	135.7	0.74	0.02	0.13
	Probable	337.0	0.62	0.02	0.12
	Total P+P	**472.7**	**0.65**	**0.02**	**0.13**
SE DOME	Proven	47.0	0.51	0.01	0.09
	Probable	69.9	0.59	0.01	0.11
	Total P+P	**116.9**	**0.56**	**0.01**	**0.10**
TOTAL	Proven	182.7	0.68	0.02	0.12
	Probable	406.8	0.62	0.02	0.12
	Total P+P	**589.5**	**0.64**	**0.02**	**0.12**
Total Ore					
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
MAIN & NW	Proven	246.4	0.73	0.15	0.13
	Probable	448.0	0.62	0.08	0.12
	Total P+P	**694.4**	**0.66**	**0.10**	**0.12**
SE DOME	Proven	57.4	0.51	0.04	0.09
	Probable	90.9	0.59	0.05	0.10
	Total P+P	**148.3**	**0.56**	**0.05**	**0.10**
TOTAL	**Proven**	**303.7**	**0.68**	**0.13**	**0.12**
	Probable	**538.9**	**0.62**	**0.07**	**0.11**
	Total P+P	**842.7**	**0.64**	**0.09**	**0.12**

Table 6-3 lists the additional Mineral Reserve held in surface stockpiles, as at June 2020.

Table 6-3 Kansanshi Mineral Reserve statement, stockpiles only as at 30th June 2020, and based on a $3.00/lb Cu price

MINERAL RESERVE AT 30th JUNE 2020 (AT $3.00/lb Cu and $1200/ounce Au)					
Oxide Ore					
S/Piles	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
SUBTOTAL	Proven				
	Probable	33.5	0.35	0.15	
	Total P+P	**33.5**	**0.35**	**0.15**	
Mixed Ore					
S/Piles	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
SUBTOTAL	Proven				
	Probable	49.8	0.59	0.20	
	Total P+P	**49.8**	**0.59**	**0.20**	
Sulphide Ore					
S/Piles	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
SUBTOTAL	Proven				
	Probable	44.8	0.40	0.02	
	Total P+P	**44.2**	**0.40**	**0.02**	
Total Ore					
S/Piles	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
TOTAL	**Proven**				
	Probable	**128.1**	**0.46**	**0.12**	
	Total P+P	**128.1**	**0.46**	**0.12**	

6.5 Production from the property

FQM production commenced in 2004 following the construction of a 4 Mtpa oxide circuit and a 2 Mtpa sulphide circuit. Commercial production was achieved in April 2005. Subsequent capital expenditure enabled an increase in the sulphide circuit capacity to 4 Mtpa, and by 2006 a second expansion had further increased sulphide production capacity 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, the various process streams 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 27.5 Mtpa.

From 2004 to December 2023, Kansanshi production amounted to approximately 4.1 M tonnes of recovered copper (Table 6-4).

Table 6-4 Copper production from Kansanshi Operations, to the end of December 2023

Year	Sulphide Feed		Mixed Feed		Oxide Feed		Total Feed		Cu in conc. Produced (kt)	Cathode Cu Produced (kt)	Total Cu Produced (kt)
	Ore (Mt)	Cu grade (%Cu)	Ore (Mt)	Cu grade (%Cu)	Ore (Mt)	Cu grade (%Cu)	Ore (Mt)	Cu grade (%Cu)			
Before 2005											80.0
2005									33.3	43.4	77.8
2006									59.2	67.7	126.9
2007									67.3	96.5	163.8
2008									101.8	113.5	215.3
2009									150.4	94.6	245.0
2010	10.38	0.77	5.46	1.29	5.67	2.26	21.52	1.29	143.0	88.6	231.1
2011	8.86	0.7	8.38	1.04	6.07	2.28	23.30	1.23	120.2	109.5	229.6
2012	9.25	0.96	8.56	1.09	6.21	2.16	24.03	1.32	155.7	104.7	260.3
2013	11.09	0.79	7.68	1.17	6.66	2.20	25.43	1.27	156.1	113.5	269.5
2014	7.94	0.93	9.41	1.13	7.98	1.77	25.33	1.27	156.6	102.4	259.0
2015	8.30	0.81	10.95	1.06	6.79	1.53	26.04	1.10	155.3	66.3	221.6
2016	11.99	0.79	7.95	1.01	7.08	1.50	27.02	1.04	172.8	79.4	252.2
2017	12.97	0.75	7.92	1.05	7.02	1.51	27.91	1.03	172.5	78.7	251.2
2018	12.98	0.78	8.19	1.06	6.92	1.44	28.08	1.02	180.0	72.4	252.4
2019	12.91	0.89	7.70	1.05	7.20	1.12	27.81	0.99	187.2	45.0	232.2
2020	13.53	0.83	8.17	1.00	7.44	0.93	29.14	0.90	169.5	52.0	221.5
2021	13.39	0.88	7.60	0.96	7.16	0.72	28.15	0.86	163.0	39.2	202.2
2022	13.16	0.71	7.71	0.63	7.87	0.57	28.74	0.65	125.7	20.6	146.3
2023	12.45	0.51	7.77	0.63	7.23	0.82	27.45	0.63	104.2	30.7	134.8
TOTAL	**159.19**	**0.79**	**113.45**	**1.01**	**97.30**	**1.45**	**369.94**	**1.03**	**2,573.7**	**1,418.6**	**4,072.8**

The first concentrate was smelted at Kansanshi in March 2015, and after a short ramp-up period, commercial production from the smelter was achieved in July of the same year. Since 2021, approximately 1.3 Mt of copper concentrate has been received into the smelter each year.

6.6 Performance relative to previous Mineral Reserve statement and production plan

Table 6-5 provides an indication of actual production performance relative to the planned production performance associated with the June 2020 Mineral Reserve statements in Table 6-2 and Table 6-3; specifically:

- actual ore tonnage and insitu copper metal depletion was 92% and 89%, respectively, of that projected over the period 2021 to the end of 2023
- waste tonnage over the same period was 84% of that projected in 2020
- hence, the total mined ore and waste was 87% of that projected
- actual stockpile movements differed from that projected, with movements to and reclaim from stockpile being 133% and 143% of that projected, respectively
- the extents of direct feed and stockpile reclaim tonnage to the plant also differed from that projected, showing figures of 62% and 138%, respectively
- with the actual lower mined ore and direct feed tonnages, and despite the higher than projected stockpile reclaim, the total plant feed tonnes (i.e., mine claim) was 91% of that projected for the period 2021 to the end of 2023
- correspondingly, the average actual copper feed grade was higher than that projected for the three year period, and hence, the total insitu copper metal was just 3% less than that projected

Table 6-5 Actual production relative to Kansanshi 2020 production schedule

	UNITS	END 2020	END 2021	END 2022	END 2023	2021 -23
MATERIAL MOVEMENTS (MINING & RECLAIM)						
PLANNED RESERVE DEPLETION		Relative to 2020 LOM Mineral Reserve schedule				
P+P PIT ORE	**Mt**	Half year in Reserves schedule	**36.7**	**40.6**	**39.8**	**117.2**
Copper grade	**%TCu**		**0.82**	**0.74**	**0.75**	**0.77**
Insitu metal	**kt Cu**		**300.0**	**301.9**	**297.9**	**899.8**
WASTE	Mt		71.1	80.2	85.2	236.5
STRIP RATIO			1.9	2.0	2.1	2.0
TOTAL TONNES	**Mt**		**107.8**	**120.8**	**125.0**	**353.6**
ACTUAL DEPLETION		Relative to 2020 LOM Mineral Reserve schedule				
P+P ORE	**Mt**	**39.2**	**40.8**	**35.5**	**30.9**	**107.2**
Copper grade	**%TCu**	**0.90**	**0.80**	**0.71**	**0.73**	**0.75**
Insitu metal	**kt Cu**	**353.3**	**325.7**	**251.2**	**225.2**	**802.1**
WASTE	Mt	57.2	64.1	69.7	65.5	**199.3**
STRIP RATIO		1.5	1.6	2.0	2.1	**1.9**
TOTAL TONNES	**Mt**	**96.4**	**104.9**	**105.2**	**96.4**	**306.5**
PLANNED STOCKPILE MOVEMENTS		Relative to 2020 LOM Mineral Reserve schedule				
Tonnes On	Mt	Half year in Reserves schedule	18.2	22.7	20.6	**61.5**
Tonnes Reclaimed	Mt		11.0	11.2	10.2	**32.4**
TONNES BALANCE	**Mt**		**140.1**	**151.2**	**161.1**	
ACTUAL STOCKPILE MOVEMENTS		Based on Production Records				
Tonnes On	Mt	28.4	31.7	24.2	26.2	**82.1**
Tonnes Reclaimed	Mt	13.7	16.9	13.0	16.6	**46.5**
TONNES BALANCE	**Mt**	**134.0**	**148.7**	**159.9**	**169.5**	
MINERAL RESERVE STATUS		Relative to 2020 LOM Mineral Reserve schedule				
P+P PIT ORE	**Mt**	**822.5**	**781.7**	**746.2**	**715.3**	
Copper grade	**%TCu**	**0.64**	**0.63**	**0.62**	**0.62**	
Insitu metal	**kt Cu**	**5,226.6**	**4,900.9**	**4,649.6**	**4,424.5**	
WASTE	Mt	3,068.0	3,003.8	2,934.2	2,868.7	
STRIP RATIO		3.7	3.8	3.9	4.0	
TOTAL TONNES	**Mt**	**3,890.5**	**3,785.6**	**3,680.4**	**3,584.0**	
PLANT PRODUCTION						
PLANNED PLANT FEED		Relative to 2020 LOM Mineral Reserve schedule				
Direct Feed Tonnes	Mt	Half year in Reserves schedule	18.4	17.9	19.3	**55.6**
Reclaim Feed Tonnes	Mt		11.5	11.6	10.7	**33.8**
Total Feed Tonnes	**Mt**		**29.9**	**29.6**	**29.9**	**89.4**
Copper grade	**%TCu**		**0.88**	**0.85**	**0.89**	**0.87**
Insitu metal	**kt Cu**		**262.0**	**251.1**	**266.0**	**779.1**
MINE CLAIM FIGURES		Based on Production Records				
Direct Feed Tonnes	Mt		11.8	11.4	11.4	**34.5**
Reclaim Feed Tonnes	Mt		16.9	13.0	16.6	**46.5**
Total Feed Tonnes	**Mt**		**28.7**	**24.3**	**28.0**	**81.1**
Copper grade	**%TCu**		**1.01**	**1.01**	**0.78**	**0.93**
Insitu metal	**kt Cu**		**289.2**	**246.6**	**219.3**	**755.1**
PLANT CLAIM FIGURES		Based on Production Records				
Total Feed Tonnes	**Mt**	**29.1**	**28.2**	**28.7**	**27.5**	**84.4**

ITEM 7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional, local and property geology

The "Domes Region" of Zambia's North-Western Province is characterised by several significant domal structures which expose large portions of the Katangan Super Group sediments. The Katangan geology is part of the Central African Copperbelt (CACB), known for its favourable copper mineralisation. Kansanshi mineralisation is located within the lower portions of the Katangan Super Group. Table 7-1 details the regional Katangan Super Group stratigraphy. Locally, the Nguba Group (previously the Lower Kundulungu) hosts mineralisation within associated metasediments. Kansanshi's surrounding regional surface geology is illustrated in Figure 7-1[2].

Table 7-1 Katangan stratigraphic column for the Zambian Copperbelt (Cyprus Amax, PFS, June 2000)

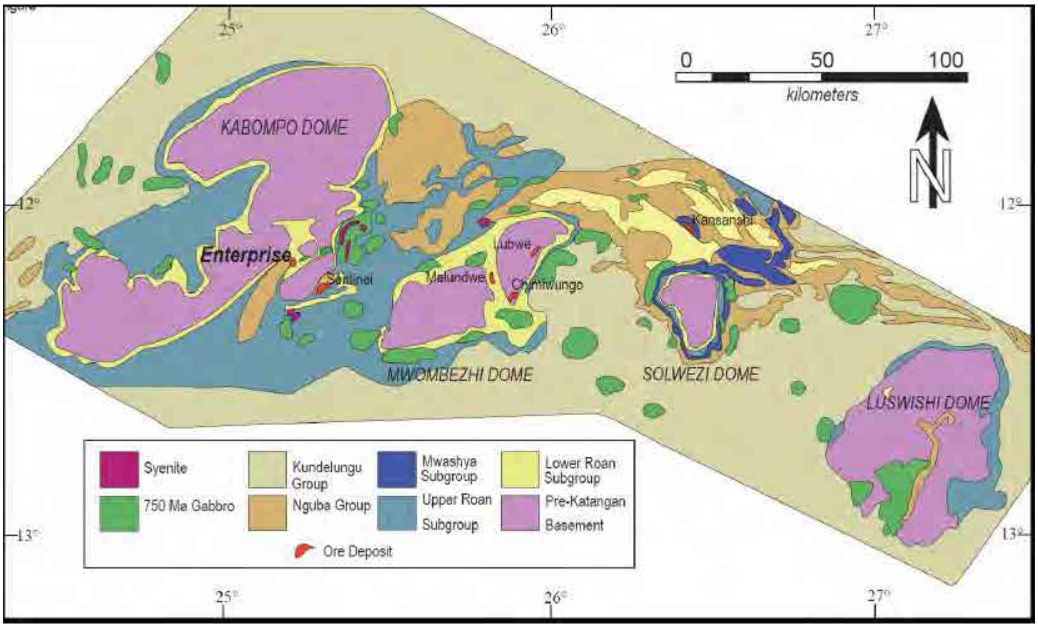
System	Series	Copperbelt Group/ Formation		Solwezi formation	Solwezi Formation (After Arthurs,1974)	Solwezi Lithology
Katanga Super Group	Lower Kundelungu	Lower Kundelungu Shale Fm	Uncertain Correlation	Solwezi Biotite-Quartz Fm	Solwezi Biotite-Quartz Fm	(calc) biotite-homblende-quartz-muse-schists, phyllite
				Upper Dolomite Fm	Chafugoma Marble Fm	
		Kakontwe Limestone				Dolomite minor phyllite
				Upper Pebble Schist Fm		
				Kansanshi Mine Fm	Pelitic Fm	Calcareous pebble schist, knotted schist, bio-muse schist; scapolite
		Grand Conglomerate/ Tillite FM		Lower Pebble Schist Fm	Paraconglomerate Fm	Marble, Calc biotite schist, locally carbonaceous phyllite knotted schist scapolite
						Calcareous pebble schist, knotted schist, bio-muse schist; scapolite
	(Copperbelt) 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

The Solwezi Dome, about 12 km south of Kansanshi (Figure 7-1), exposes much of the Katanga Super Group which is mostly comprised of granites, migmatites and gneisses. The area around the Solwezi Dome consist of metamorphosed schists, quartzites and conglomerates, believed to be chronological equivalents of the Lower Roan Group, a characteristic unit within the Zambian Copperbelt (Cyprus Amax, 2000). To the north of the Solwezi Dome, banded ironstones, quartzites and phyllites are believed to represent the Mwashia Group (Arthurs, 1974) (Figure 7-1).

Calcareous-dolomitic and quartz-biotite-hornblende-garnet schists are found between the rocks surrounding the Solwezi Dome and Kansanshi (Figure 7-2). Drilling, geological interpretations and modelling suggests that these schists overlie the Kansanshi Mine Sequence which implies an antiformal folding with later transposition of the Katangan-aged metasediments. The axis of the Kansanshi Antiform trends in a NW-SE direction.

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

Figure 7-1 Simplified regional geology of the "Domes Region" of North-Western Province, Zambia (Capistrant et al., 2014)



Further deformation along the strike of the Kansanshi Antiform (Figure 7-3) has resulted in the formation of double plunging domal structures along its crest. These domed structures host the three Kansanshi deposits known as the North West, Main and South East deposits. Table 7-2 presents a detailed summary of the local Kansanshi Mine Stratigraphic Sequence.

For a comprehensive description of the geological, structural, and tectonic setting of Kansanshi, reference can be made to the Cyprus Amax PFS (2000), to Beeson et al. (2008), and to MacIntyre (2019).

Figure 7-2 Simplified local geology of the Kansanshi Deposits, Solwezi, North-Western Province, Zambia (MacIntyre, 2019)

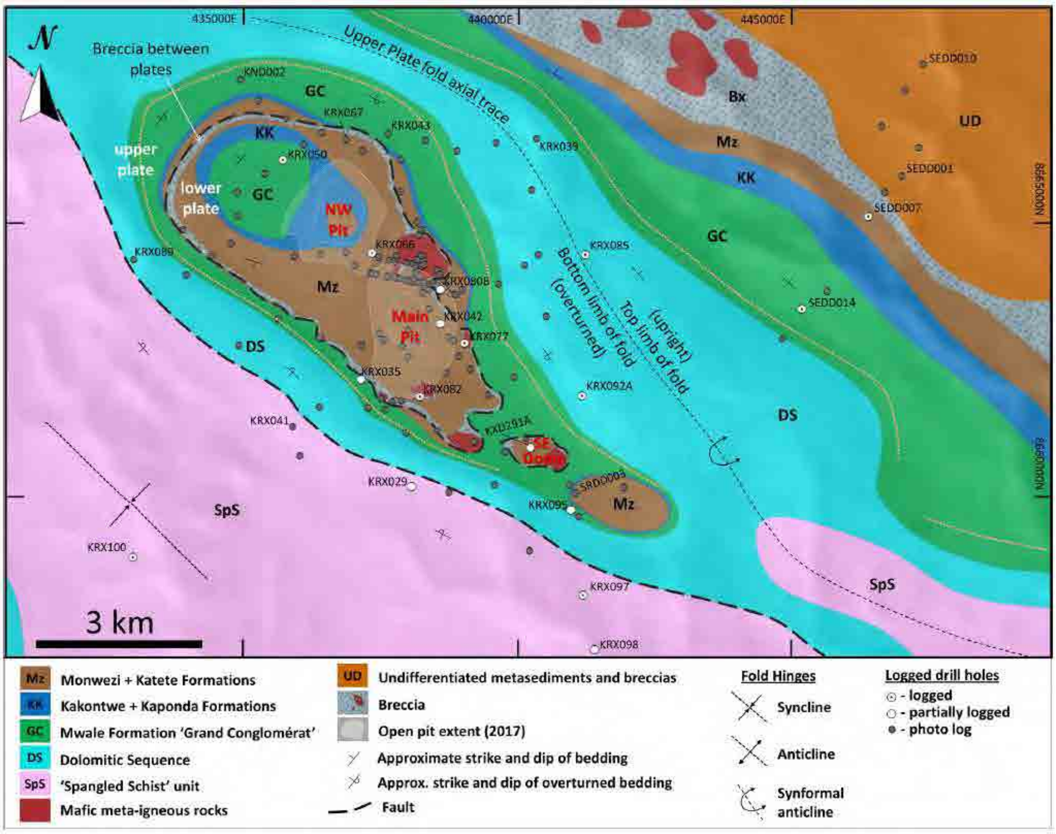


Table 7-2 Kansanshi Mine Stratigraphics Sequence (MacIntyre, 2019)

Group	Kansanshi Units	Historic Mine Stratigraphy	Thick. (m)	Facing
Roan	Spangled Schist	Mica Schist	~600+	Overturned Sequence
Roan	Dolomitic Sequence	Upper Dolomite	250-375	Overturned Sequence
Nguba	Mwale Formation	Upper Pebble Schist	100-220 <50 to 300+	Overturned Sequence
Nguba	Kakontwe Formation	Top Most Marble	0 - 20	Overturned Sequence
Nguba	Monwezi Formation	Upper Mixed Clastics	0-70	Overturned Sequence
Roan	Evaporitic Sequence	Meta-gabbro	0-280+	
Nguba	Monwezi Formation	Upper Mixed Clastics	100-150 <10 to 200+	Folded repeated sequence
Nguba	Kakontwe Formation Mwale Fm	Upper Marble Middle Pebble Schist Upper Marble	40-90 <30 to 170+	Folded repeated sequence
Nguba	Monwezi Formation	Middle Mixed Clastics Cu	30-80	Folded repeated sequence
Nguba	Katete Formation	Lower Calcareous Sequence	10-40	Upright Sequence
Nguba	Kakontwe Formation	Lower Marble	60-90 <20 to 180+	Upright Sequence
Nguba	Mwale Formation	Lower Pebble Schist	120-220 <30 to 480+	Upright Sequence
Roan	Dolomitic Sequence	Lower Dolomite	60-300	Upright Sequence
Roan	Evaporitic Sequence	Upper Roan	270-800	Upright Sequence
Roan	Basal Clastic Sequence	Lower Roan	>1200m	Upright Sequence

7.2 Structure

Structural deformation of the Kansanshi Mine sequence lithologies has led to the formation of domal-shaped geometries along the antiform axis, creating favourable traps for the Kansanshi mineralisation. The mineralisation within the North West, Main and South East deposits is localised within these domes along the crest of the overall Kansanshi antiform (Figure 7-3, Figure 7-4 and Figure 7-5).

The strongest mineralisation occurs near the apex of these domes and diminishes with increasing distance and depth from these points. The shallower dome apexes, closer to the surface, have had strong exposure to supergene processes, resulting in enriched copper mineralisation. Mineralisation decreases rapidly with depth from the base of the Middle Mixed Clastics (MMC) unit.

The Main deposit has two main structural zones, known as the 4800E and 5400E, oriented north-east to south-west. These zones are characterised by faulting, veining, and brecciation, with deeper weathering and oxidation contributing to relatively enriched copper mineralisation. While the North West and South East

Dome deposits also experience varying degrees of structural deformation, it is less pronounced compared to the Main deposit.

Figure 7-3 Perspective view of simplified schematic deformation model of Kansanshi's Main and North West deposits (FQM Exploration Team, Domes Project Review, 2024)

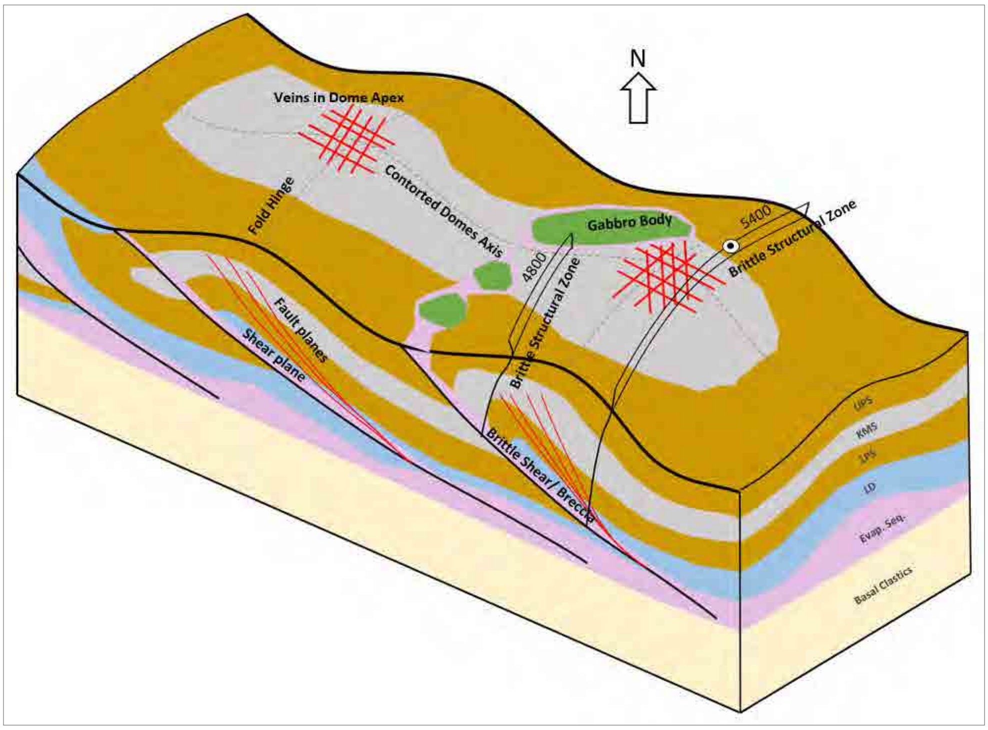
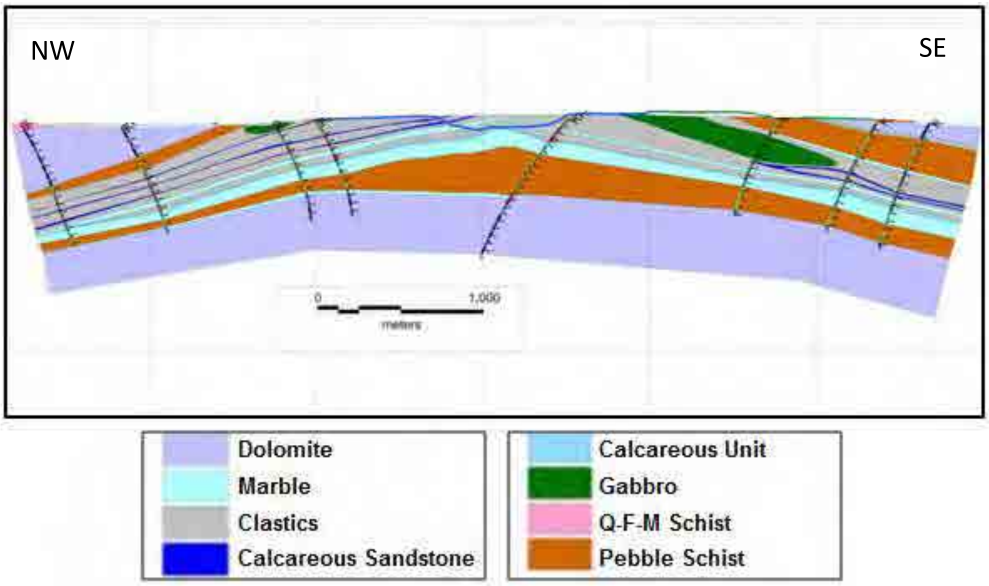


Figure 7-4 A cross section of the Kansanshi antiform illustrating prevailing geology (FQM, 2024)

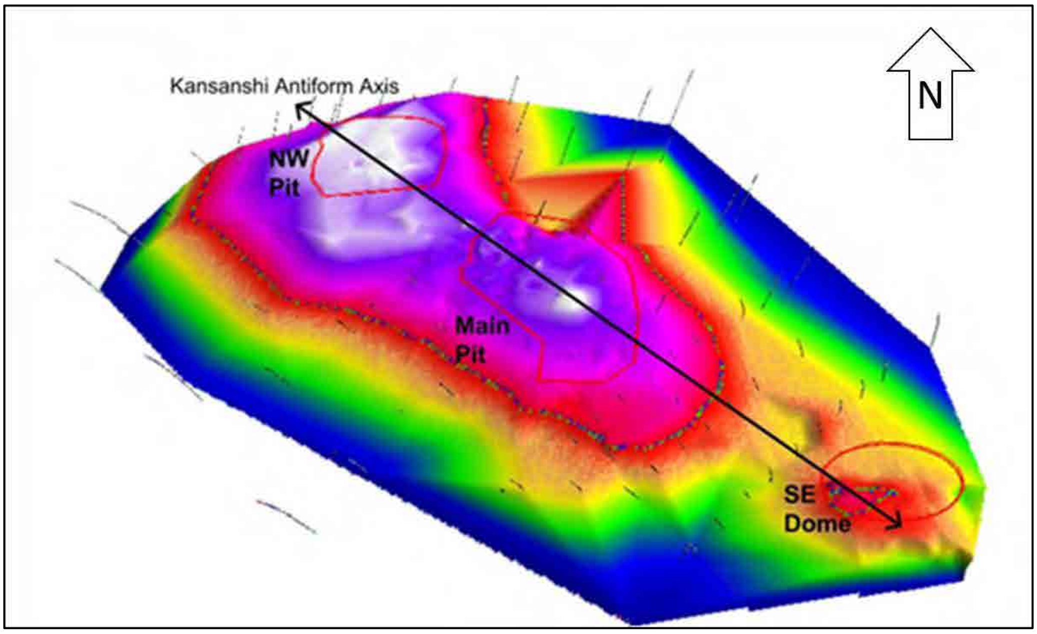


An important structural feature of the Kansanshi deposits is the presence of well-developed vein networks. Structural deformation has facilitated late post metamorphic and undeformed high-angle quartz-carbonate veins across all three deposits. The orientations of these veins are influenced by the structural framework and the dome morphology, resulting in distinct patterns at each deposit. In summary:

- The North West deposit exhibits four steeply dipping vein sets that form a mesh-like geometry, widening as they extend away from the dome apex.

- The Main deposit has steep dipping, predominant north-south oriented vein sets with additional flat-lying veins aligned sub-parallel to the Lower Marble lithologies. Smaller scale veins radiate from the dome apex.
- The South East deposit has steeply dipping NW-SE oriented vein sets, with frequency decreasing with distance from the dome apex.

Figure 7-5 Elevation contours of the Upper Marble, showing the apex for each of the North West, Main and South East deposits, and the antiform axis orientation (FQM, 2024)



7.3 Styles of mineralisation

Mineralisation across each of the Kansanshi deposits occurs as shallower oxides and secondary copper minerals, with deeper underlying primary copper sulphides. Mineralisation is influenced by depth below surface, distance from the dome apexes, structural features (faulting, veins, and breccias) and lithological associations.

7.3.1 Stratabound mineralisation

Stratabound mineralisation hosts the majority of Kansanshi's copper metal. Bedding-parallel disseminated chalcopyrite is dominant within phyllites and carbonaceous phyllites (Figure 7-6) with a tendency for stronger mineralisation closer to larger veins. Organic carbon within the phyllites acts as a reducing agent, facilitating the precipitation of sulphides along specific strata horizons. Chalcopyrite is the dominant sulphide species, accompanied by lesser amounts of bornite and molybdenite, and with increasing pyrite and pyrrhotite more distal to the stronger chalcopyrite mineralisation. Oxide minerals (malachite, chrysocolla) and secondary sulphides (chalcocite) are more abundant in the shallower weathered and oxidised zones.

Figure 7-6 Stratabound and stringer veinlet mineralisation within a carbonaceous phyllite

7.3.2 Vein mineralisation

Vein mineralisation hosts more than 30% of total copper metal across the three deposits. The veins are primarily composed of quartz/carbonate with accessory copper minerals (Figure 7-7), typically exhibiting sub-vertical orientations and ranging from centimetre to metre-scale widths. Chalcopyrite is the principal copper sulphide mineral while malachite, chrysocolla, and chalcocite are common in the shallower, more weathered parts of the veins (Figure 7-8). Veins are common within the clastic units but may extend into adjacent marble units. Concentration of vein mineralisation tends to be higher in the contact zone with its host rock.

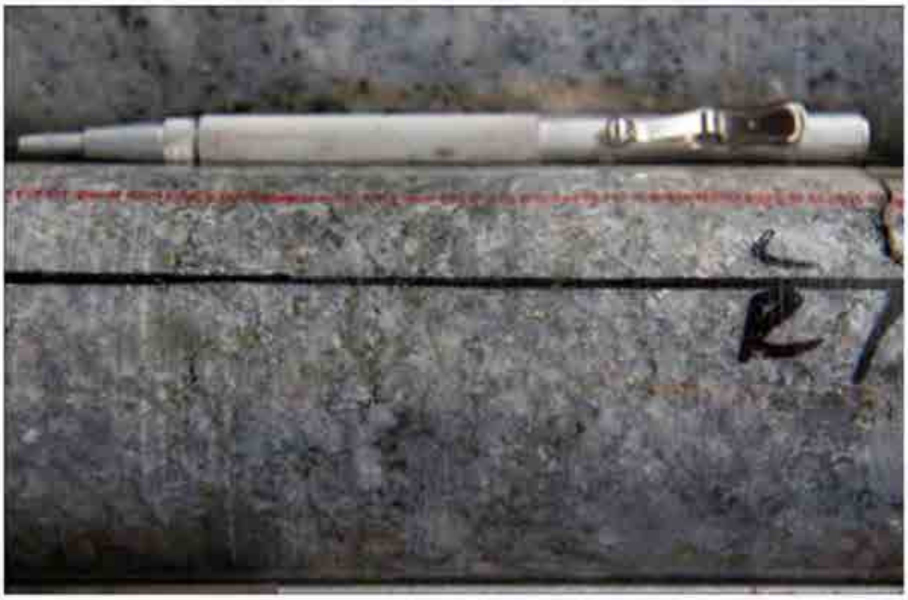
Minor vein constituents such as anhydrite, magnetite, and trace uraninite are occasionally present. Molybdenite is notably common in calcite veins at the Main and NW deposits.

Figure 7-7 Vein hosted copper mineralisation in a fresh sulphide zone at Kansanshi

Figure 7-8 Malachite, chalcocite assemblages within oxidised calcitic veins

Mineralised veins often have albite alteration halos that bleach the immediate host lithologies. These halos vary in thickness from millimetres to several metres, depending on lithology and vein width. These alteration halos are larger and more diffuse within carbonaceous units but more restricted within marbles and schists. Disseminated mineralisation is often associated with these halos and is common in phyllites, carbonaceous phyllites and marble units (Figure 7-9).

Figure 7-9 Disseminated mineralisation seen in marble within the sulphide zone



7.3.3 Breccia

Breccia mineralisation occurs where there has been brittle structural deformation, creating space to trap sulphide mineralisation. Although less common at Kansanshi, brecciated zones can exhibit strongly mineralised stockworks along with increased veins occurrences (Hanssen et al., 2010). Breccia mineralisation is most common in the Main deposit, also with localised breccia mineralisation associated with gabbroic intrusives and the 4800E and 5400E structural zones.

7.4 Types of mineralisation

Copper mineralisation at Kansanshi comprises oxide copper, secondary copper sulphide and primary copper sulphide minerals together with a range of mixing thereof. Minor refractory copper is present within sub-surface portions of the saprolite.

The relative proportions and degrees of mixing of these copper minerals vary across each deposit and are influenced by the prevailing geology. Mineralisation generally favours clastic over carbonate lithologies. Sulphide mineralisation consist primarily of chalcopyrite, with minor bornite, and some gangue pyrite and pyrrhotite.

Primary and secondary copper sulphide minerals are finely disseminated within strata horizons, becoming coarser nearer to and within the veins. The Upper Mixed (UMC) and Middle Mixed Clastic (MMC) units host most of the veins and stratabound mineralisation.

7.4.1 Refractory

Refractory mineralisation often outcrops and is mostly located immediately below the surface within the saprolite zone of the Upper Mixed Clastics. Copper is contained within a clay matrix of smectite, iron oxide and hydroxide minerals. Copper is not recoverable by normal processing methods from refractory copper minerals and is excluded from Mineral Resources.

7.4.2 Oxide mineralisation

Most oxide copper mineralisation occurs closest to surface and is often semi-massive. Weathering and oxidation of primary sulphides has led to zones of enriched supergene copper oxide and secondary chalcocite mineralisation. The presence of oxide and secondary copper minerals is typically strongest at the apexes of the domes in the North West and Main deposits. In contrast, South East deposit has relatively underdeveloped oxide and secondary copper mineralisation.

Oxide mineralisation, primarily malachite, tenorite, chrysocolla, and minor azurite, occurs within veins, alteration haloes, faults, fractures, breccias and strata. Oxide mineralisation is dominant in the shallowest portions of deposits and is influenced by morphology, affecting the mobilisation and dispersion of copper minerals along complex weathering pathways. Weathering tends to be strongest along and down vertical structures and vein sets. As a result, oxidised mineralisation can occur at depths well below the average depth of weathering. Gangue minerals consist mainly of smectite clays, calcite, marble, iron oxides and hydroxides.

7.4.3 Mixed mineralisation

Mixed mineralisation forms broad transition zones between oxide and underlying sulphide mineralisation. These transition zones contain mixtures of primary copper sulphides (chalcopyrite), secondary copper sulphides (chalcocite) and copper oxide minerals. The ratio of single acid soluble copper to total copper is useful for determining the degree of mixing of these respective minerals and therefore which processing circuit will yield optimal copper recovery.

7.4.4 Sulphide mineralisation

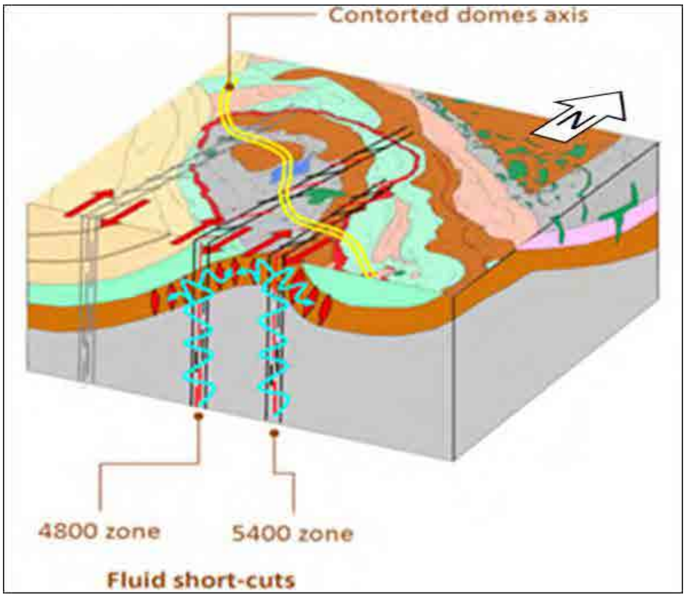
Copper mineralisation across Kansanshi is mostly fresh (unoxidised) primary copper sulphides (chalcopyrite) with rare isolated occurrences of bornite. Other common primary sulphides include pyrite and pyrrhotite. Primary sulphide mineralisation is typically located below the oxidised and mixed mineralisation zones and is associated with both veins and along select strata horizons.

ITEM 8 DEPOSIT TYPES

The Kansanshi copper mineralisation is believed to have originated from hydrothermally remobilised copper controlled by lithological and structural factors. Oxidised copper-enriched hydrothermal fluids are thought to have originated from magmatic sources, migrating upwards and acquiring significant mineral and metal content by leaching and scavenging elemental copper and gold from basement and/or overlying sedimentary rocks (Figure 8-1). Copper mineralisation across the Kansanshi deposits was precipitated within dome-shaped trap structures under prevailing reducing conditions within the host lithologies. Sulphide and associated metal deposition occurred at redox interfaces where oxidised copper-enriched fluids intersected reducing carbonaceous sediments such as the carbonaceous phyllites. The interaction between graphitic organic matter and hydrothermal fluid likely controlled the oxidation state of the ore fluid, leading to mineralisation.

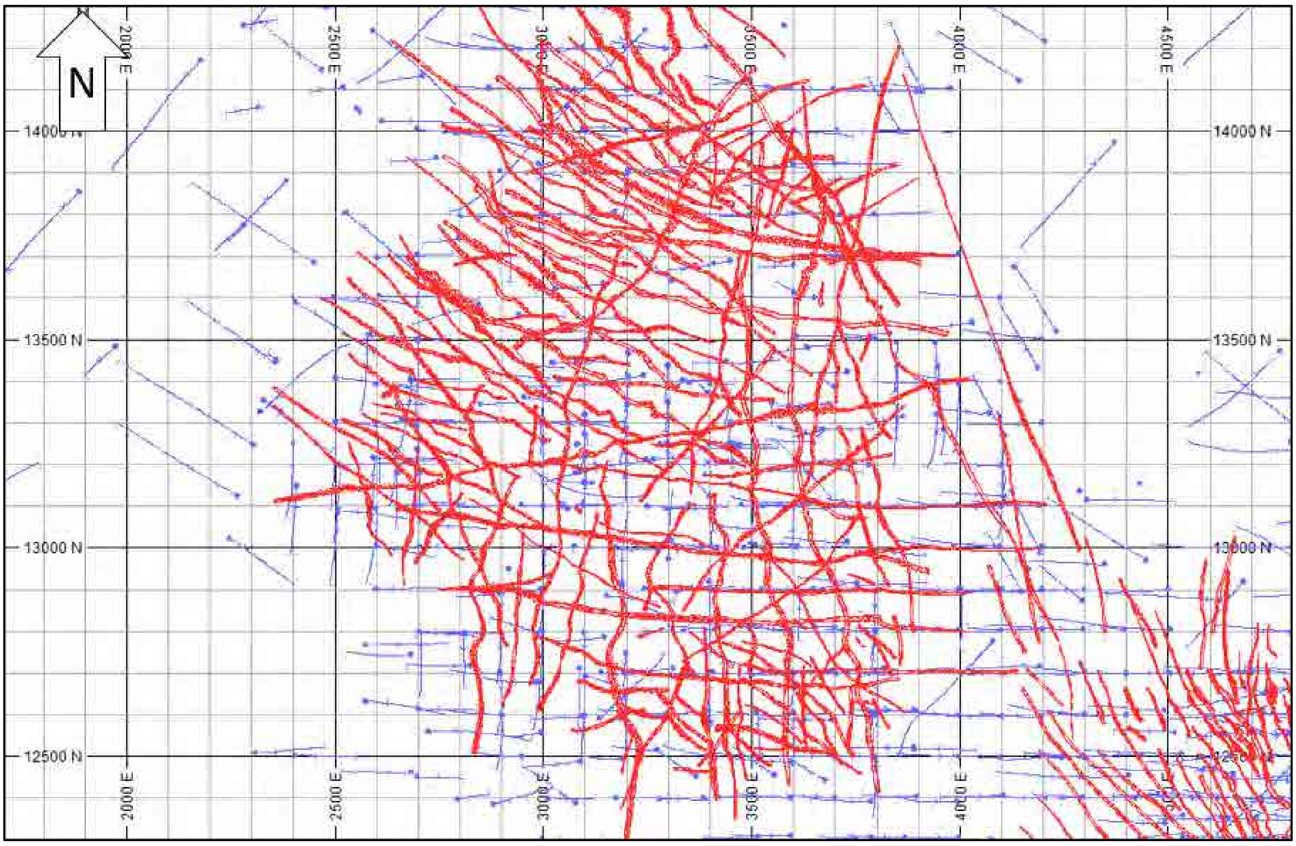
The 4800E and 5400E structural zones exemplify potential fluid pathways. Mineralisation traps are clearly concentrated around the immediate apexes of domes, favouring specific lithologies as well as structural features and veins. This geological framework has informed the modelling approach and exploration work, particularly with multi-directional holes drilled at approximately 60° to 70° to maximise intersection angle with the antiformal shape of the lithological units and veins, thereby accounting for multiple vein and fold orientations (Figure 8-2). This approach optimises the angle of intersection, reducing the risk of sample assay results distorting the dimensions of true mineralisation volumes.

Figure 8-1 Genetic model showing possible pathways for hydrothermal fluid migration (FQM, 2024)



Initial drilling efforts focused on defining the shape and extent of the dome structures, with holes drilled at approximately 100 m grid spacing, oriented both N-S and E-W. Sectional interpretations of the drilled logging data guided infill drilling to a 25 m to 50 m square grid, with closer spacing in better mineralised dome apex areas and increasing outward to the extents of the mineralisation. Continuous pit mapping has supplemented geology definition at a scale finer than the drill grid spacing.

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 (FQM, 2024)



Geological modelling has focused on key aspects of the style of deposit mineralisation through 3D modelling of vein volumes, key stratigraphy and internal lithologies hosting the mineralisation. The resulting 3D models are utilised to delineate key domains and controls used for Mineral Resource estimation. Strata-controlled mineralisation has flatter-longer ranges of continuity in contrast to veins, which tend to display more sub vertically oriented continuity.

ITEM 9 EXPLORATION

9.1 Introduction

Exploration at the Kansanshi Mine lease and adjoining properties has been active since the early 1950s, conducted by Anglo American, ZCCM, Cyprus Amax and FQM. Exploration techniques include surface geological mapping, soil sampling, geophysics, and both diamond and RC drilling.

9.2 Geophysical surveys

Geophysical surveys acquired between 1997 and 2023 aimed to enhance the detection of copper mineralisation by mapping host geological structures, understanding deposit characteristics, and assisting in drill targeting. Techniques included airborne electromagnetics (AEM), downhole geophysics, audiomagnetotellurics (AMT), induced polarisation (IP), aeromagnetics, radiometrics, seismic and gravity surveys. The acquisition areas are shown in Figure 9-1, and the work completed is summarised in Table 9-1.

Magnetics, AEM and AMT were the most effective in defining and targeting host lithologies and domal structures, while the CSAMT, SAM and IP were aimed at directly targeting mineralisation with variable success. IP supported the AEM results in defining the carbonaceous phyllite host unit but was less effective at directly targeting veins or mineralisation. Ground gravity helped refine lithological boundaries and structures. Since 2010, downhole geophysical surveys (including sonic, density, magnetic susceptibility, IP and gamma) on 12 drill holes have improved the understanding of the physical properties of the Kansanshi lithologies.

Most recently, in 2023, AEM surveys were acquired to the east and northeast of the mine to as part of a new brownfields campaign targeting new domal structures along strike and adjacent to the Kansanshi dome. This program is still underway, with 11 diamond holes drilled in 2023 for a total of 5,243 m.

Figure 9-1 Map of areas covered by geophysical surveys from 1997 – 2023 (FQM, 2024)

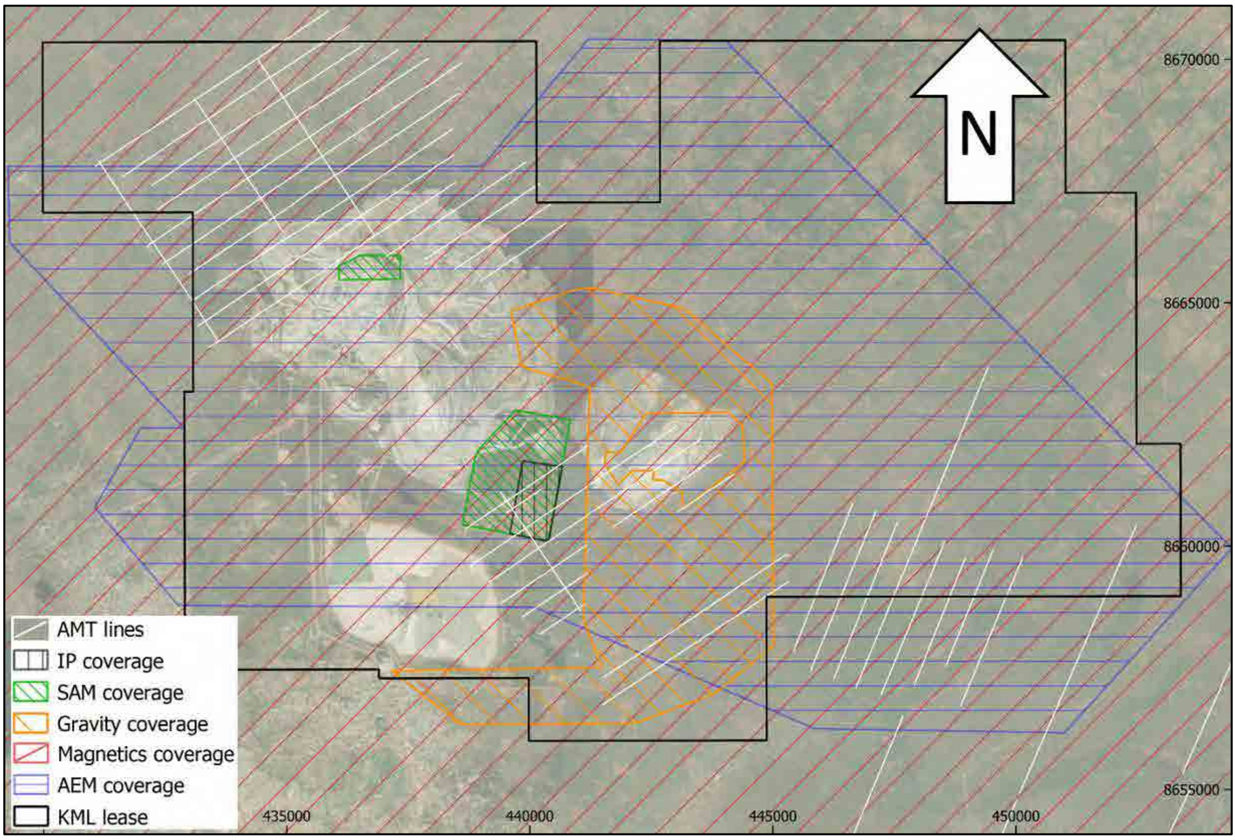


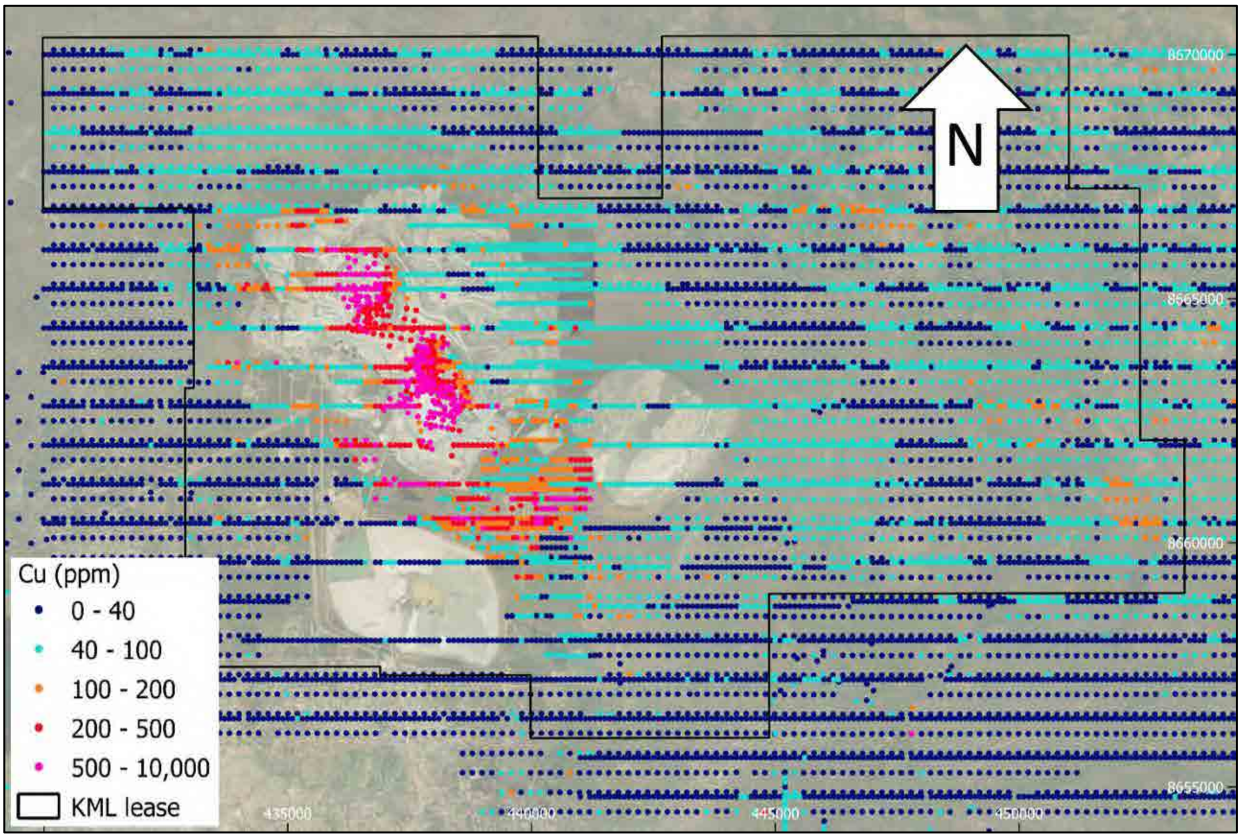
Table 9-1 Geophysical campaigns (by method) completed at Kansanshi

Method	Year	Project Area	Service Provider/ Contractor	Description
Sub-Audio Magnetics (SAM)	2011	North of North West (NW) Open Pit	GAP Geophysics (Australia)	25m spaced sub-audio magnetics for near-surface resistivity mapping.
		South East Dome Deposit		
Gravity	2014	East of Kansanshi Mine	Engineering and Exploration Geophysical and survey (EEGS)	100m x 100m stations ground gravity survey east of Kansanshi Mine.
Airborne Electromagnetics (EM)	2007	Kansanshi dome	Fugro Airborne Surveys	107 km^2 of Kansanshi Mine Lease
	2023	Southeast Dome and eastern extents of mine lease	New Resolution Geophysics	200 m line spacing covering 140 km^2
Airborne Magnetics (Aeromagnetics) and Radiometrics	1997	Solwezi Dome and Kansanshi Deposit	High-Sense Geophysics	200m spaced lines - Survey covering both Solwezi Dome and the Kansanshi Deposit
	2010	Kansanshi Mine - excluding active mining area	New Resolution Geophysics (NRG)	118 km^2 at 100m spaced lines - High Resolution Survey covers 118km^2 of Kansanshi Mine Lease (also radiometrics), excludes the active mining area.
	2014	South West corner of Kansanshi Mine Lease		100m spaced lines - Survey covers South West corner of the Kansanshi Mine Lease.
Induced Polarisation (IP)	2010	South East Dome Deposit	Geophysical Surveys and Systems (GSS)	1.2km^2 over South East Dome
Seismics	2013	South East Dome Deposit	Uppsala University	3 x seismic lines (19.3 km) - 2 x lines over South East Dome and 1 x line north of North West Pit.
		North of North West (NW) Open Pit		
Controlled Source Audio-magnetotellurics (CSAMT)	2009	South West of Main Pit	Geophysical Surveys and Systems (GSS)	2.5 line km (CSAMT) - 4 x CSAMT lines completed to the South West of Main Pit
Natural Source Audio-magnetotellurics (NSAMT)	2011	South East Dome Deposit	Geophysical Surveys and Systems (GSS)	1.2km (NSAMT) trial survey over South East Dome
	2012	North of North West (NW) Open Pit		67.5 line km (NSAMT) - North of North West Pit.
	2014	South East Dome Deposit		81.8 line km (NSAMT) - over South East Dome
	2018	Solwezi East		50.5 line km (NSAMT) - over Solwezi East

9.3 Soil sampling

Surface geochemical soil sampling has principally been used as a mineralisation vectoring tool to prioritise exploration prospects and generate drill targets. Since 1,997, extensive soil sampling campaigns have been completed over the Kansanshi lease. Systematic grids were collected in 2004 and 2014, with most of the mine licence area covered by samples spaced 100 – 200 m on 400 m spaced lines. Assay methods included aqua regia in 2004 and ME-MS61 analysis in 2014 on homogenised samples. Focussed, tightly-spaced surveys were also conducted to infill previous grids and follow up geophysical anomalies. These sampling campaigns, combined with geophysics, have helped with the litho-geochemical characterisation of the deposit area, surface geology mapping of the mine licence and regolith domaining. The locations of these grids are shown in Figure 9-2.

Figure 9-2 Maps of the historic and recent soil sampling coverage on the Kansanshi Lease (black outline) (FQM, 2024)



These exploration soil samples were not part of the Mineral Resource Estimates included in this Technical Report.

ITEM 10 DRILLING

Kansanshi Mineral Resource estimates are supported by geology and mineralisation data collected from diamond drilled (DD) core samples and reverse circulation (RC) drilled chips. These samples were geologically logged, sampled and analysed for metal and element concentrations. The resulting data was then used to create 3D geology models and block model estimates of metal grades. Diamond drilling provides broader deposit coverage with deeper holes, albeit at wider grid spacings compared to RC drilled holes. RC drilling provided detailed infill geology and mineralisation data to improve the local accuracy of mineralisation location, shape, volume, tonnage and metal grades. This detailed RC drilled data supports estimates used for short-term planning and ore control. Both diamond and RC drilling data are used for resource estimation.

10.1 Diamond drilling

Since the 2020 Technical Report, diamond drilling has continued, focussed on the Main and South East deposits (Figure 10-1). An additional 209 DD holes and 20,561 RC holes were added from September 2019, the cut-off date for the drilling data included in the previous Technical Report. The key objectives have been to extend and close-out the deposit mineralisation, and to infill wider grid areas. These drilling programmes were planned and managed by KMP's Mine Geology team.

Figure 10-1 Kansanshi diamond drillhole coverage with new holes (2019-2024) in green (FQM, 2024)

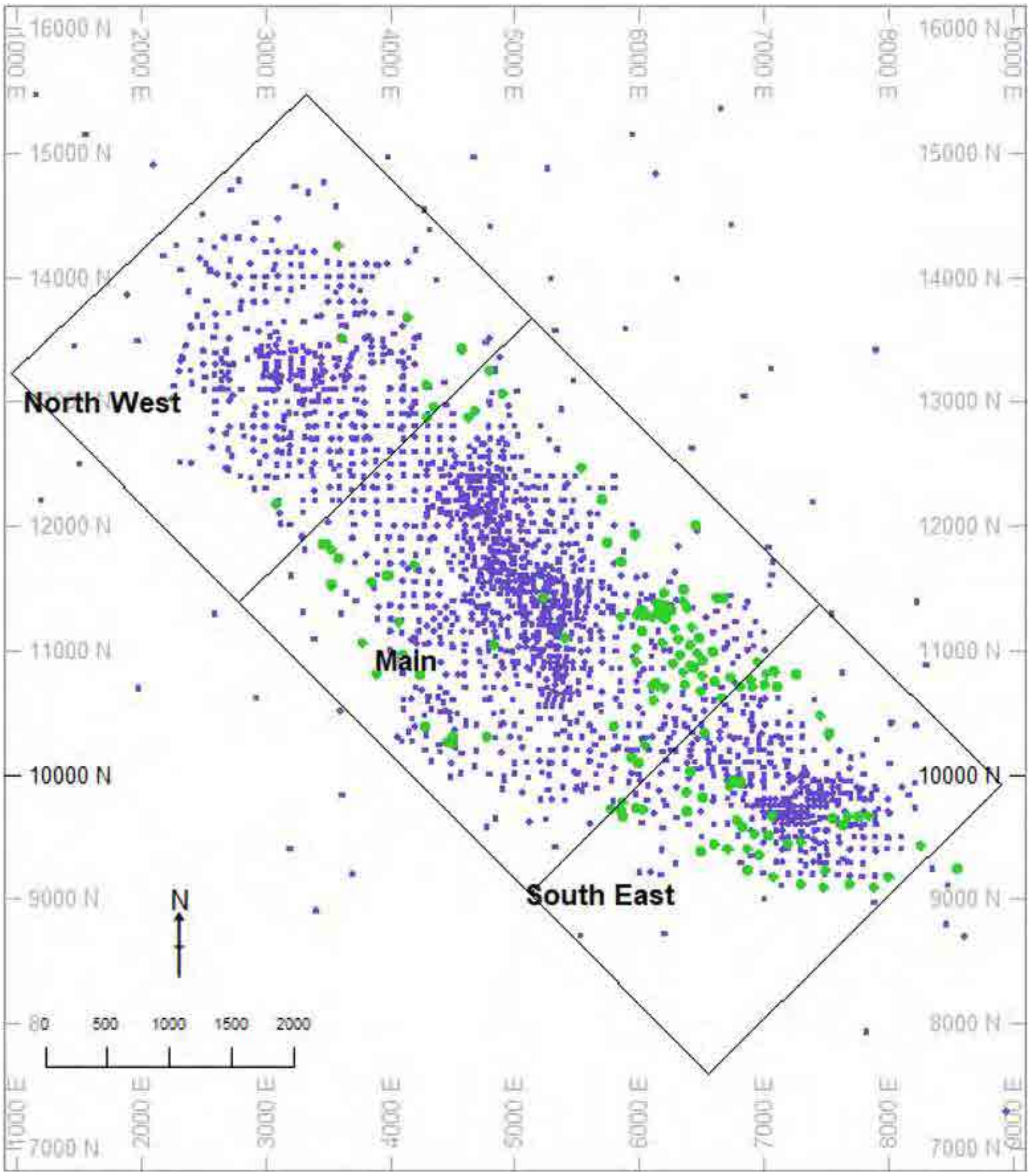


Table 10-1 is a summary of the additional diamond and RC drilling undertaken since the 2020 Technical Report. Historic campaigns are summarised in Table 10-2.

Table 10-1 Summary of additional drilling and sampling since the 2020 Technical Report

Hole_Type	Count_Holes	Sum_Metres	Count_Samples
RC	20,561	1,296,301	426,794
DD	209	63,838	27,412

Table 10-2 Kansanshi drill campaign summary

Year	Company	Reverse Circulation (RC)		Diamond Drilling (DD)		Totals (all)	
		RC _Holes	RC_Metres	DD_ Holes	DD_ Metres	Total_Holes	Total_Metres
1980	ZCCM	-	-	99	16,823	99	16,823
1997	CPA	7	935	4	123	11	1,058
1998		27	4,212	312	66,535	339	70,747
1999		-	-	71	15,035	71	15,035
2001	KMP	66	5,880	33	4,131	99	10,011
2003		44	3,250	-	-	44	3,250
2006		-	-	31	4,969	31	4,969
2007		886	41,012	41	5,974	932	47,526
2008		1,160	31,571	31	7,236	1,191	38,807
2009		2,074	63,192	151	39,769	2,225	102,961
2010		1,668	78,698	150	45,302	1,818	124,000
2011		3,508	181,213	319	112,056	3,827	293,269
2012		4,662	272,349	329	149,520	4,991	421,869
2013		6,657	381,420	203	98,644	6,860	480,064
2014		5,843	310,503	52	29,755	5,895	340,258
2015		5,940	322,641	-	-	5,940	322,641
2016		7,623	487,380	32	10,733	7,655	498,113
2017		5,657	387,002	43	16,722	5,700	403,724
2018		5,358	380,736	61	27,521	5,419	408,257
2019		3,462	240,066	44	20,525	3,506	260,591
2020		4,362	339,342	54	24,909	4,416	364,251
2021		4,646	263,484	31	14,134	4677	277618
2022		6,811	346,200	52	13,652	6863	359852
2023		708	36,027	36	3,507	744	39534
Total		**71,169**	**4,177,113**	**2,179**	**727,575**	**73,353**	**4,905,228**

Diamond drilling was conducted at various grid spacings, with large areas covered by grids less than 50 m. The primary focus was on defining the extents and continuity of geology and mineralisation at each deposit. Given the sub-horizontal strata with multiple oriented sub-vertical veins, several drilling directions at dips between 60° to 90° were employed to ensure comprehensive geological coverage and detailed sampling of stratabound and vein-style mineralisation. Drilling directions and angles were guided by the prevailing antiform strike and dip, the location relative to structural domains, and the dominant vein orientation known from pit mapping, maximising the angle of intersection with vein and strata mineralisation.

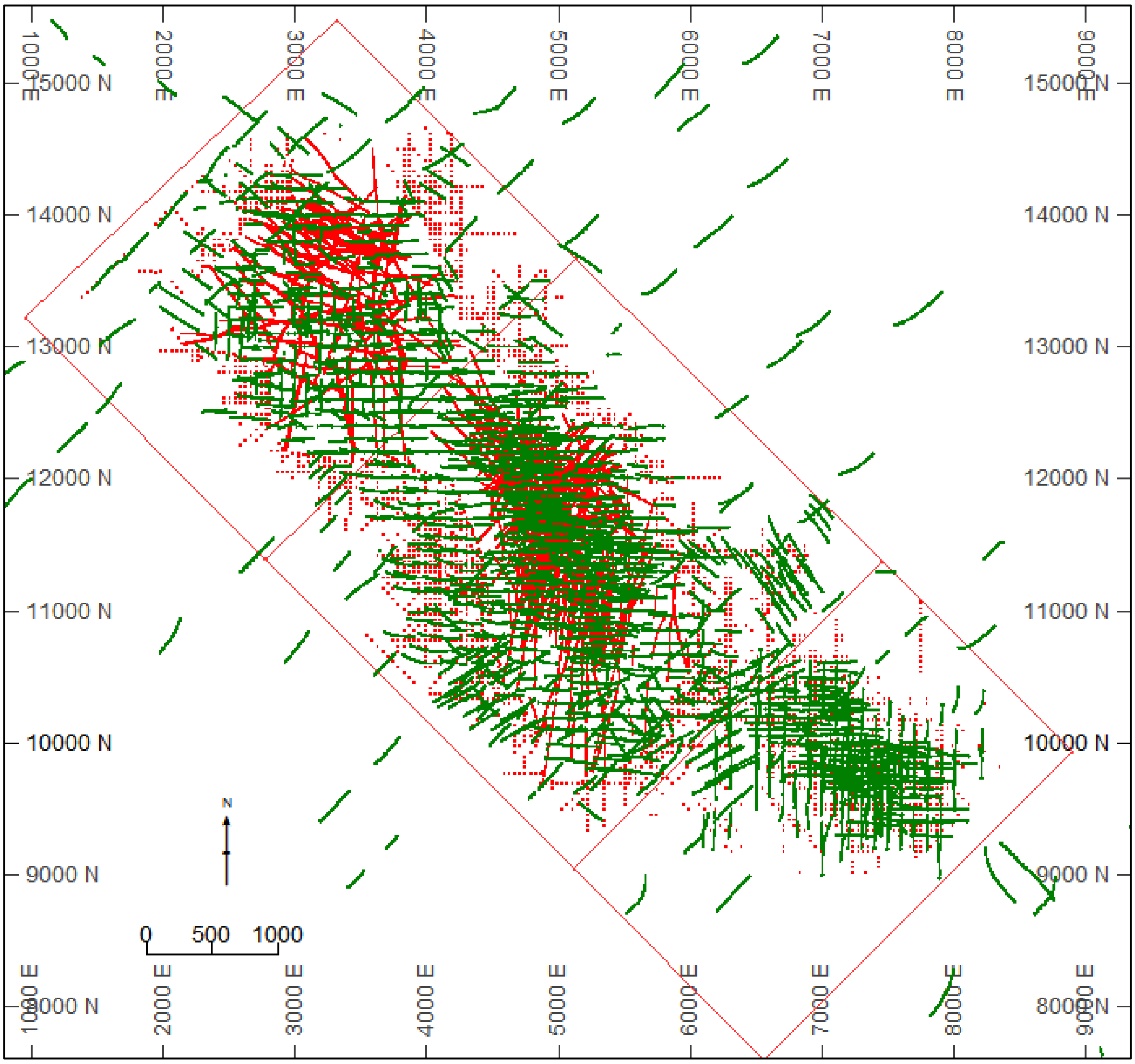
At the Main deposit, veins generally trend in N-S and NNE-SSW directions. As such, drilling in the Main pit area was oriented E-W to achieve the highest intersection angles with N-S veins and structural domains. The Main deposit features a higher grade core where stratigraphic units are mineralised down to the Lower Pebble Schist. Several in-pit diamond holes, drilled in 2017 and 2018 intersected mineralisation.

The North West deposit has three dominant vein orientations: N-S, E-W and NW-SE. Drilling in N-S and E-W directions was required to intersect veins at a maximum angles. Regional drill holes were oriented in N-E direction, perpendicular to the Kansanshi Antiform (Figure 10-2).

Drilling at the South East deposit has focused on a regular grid of holes drilled E-W and N-S to best define the shape and extent of the dome. Holes were drilled at 100 m to 50 m grids, increasing with distance from the centres of the deposits.

Sample lengths ranged from 0.5 m to 3.0 m, aiming to honour geological and mineralisation boundaries where practical. Most mineralised veins are wider than one metre, while strata mineralisation widths range from a few metres to tens of metres.

Figure 10-2 Drill orientations across Kansanshi deposits. Veins are coloured in red and hole traces in green (FQM, 2024)

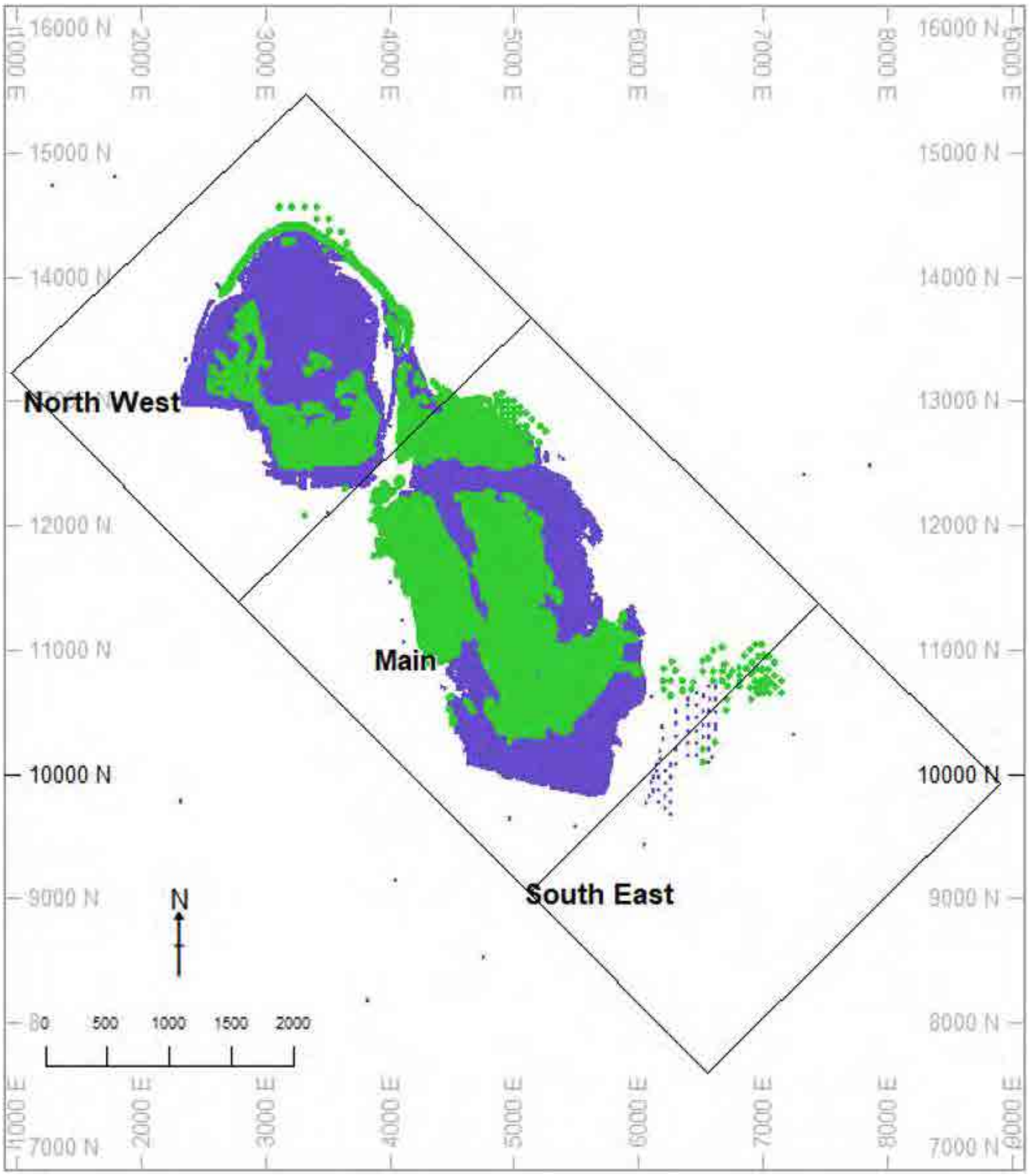


10.2 Grade control Reverse Circulation (RC) drilling

RC drilling within the pits follows a grid of approximately 10 m (north) by 12.5 m (east), with holes dipping at 60° and drilled to lengths of 51 m to 72 m. This provides vertical coverage between 40 m to 60 m, equivalent to four to six 10 m high mining benches. The azimuths vary between 90°, 220° and 270°, depending on the prevailing geology and the need for the highest intersection angles. RC holes were drilled to depths of 150 m southeast of the Main deposit to improve definition in the oxidation transition zone, and to inform planned mining at the South East deposit. RC drill chips were sampled and logged to provide detailed geology and assay grades for short-term planning and ore control. A comprehensive QAQC programme ensured accurate and precise sample results, with demonstrated control on contamination for the RC samples. Coordinated RC drill holes and QAQC results support the use of RC logging and sample assay results in this estimate.

The available RC dataset consists of a total of 71,169 drilled RC holes representing 4,177,113 m and 1,677,312 samples (Figure 10-3).

Figure 10-3 RC ore control drilling coverage showing holes drilled between late 2019 to 2024 in green (FQM, 2024)



10.3 Core recovery analysis

Core recovery and RQD data are routinely recorded during drilling and logging. Drilling lengths with poor recovery may not adequately represent the prevailing geology and mineralisation and may therefore need to be excluded from the estimation routine. Recovery data was analysed by drilling contractor, weathering and stratigraphy. Overall, core recovery is above 90%, posing a very low risk to block model estimates. Analysis of core recovery by drilling contractor has highlighted no significant biases (Table 10-3).

Table 10-3 Core recovery by contractor

Contractor/Recovery %	
Tectonic Drilling Services (TECT)	92.6
Titan Drilling Services (TDS)	90.7

Analysis of recovery by core diameter confirms lower expected recoveries in weathered material (Table 10-4 and Table 10-5). Soils and laterites tend to lower recovery values. Comparison of core recovery to metal grades suggests limited to no impact of recovery on analysed grades.

Table 10-4 Core recovery by core diameter

Core Diameter/ Recovery %	
HQ	95.6
NQ	91.8
PQ	80.8

Table 10-5 Core recovery by weathering[3]

Weathering/ Recovery %	
SOIL	62.2
LAT	63.1
SAP	76.9
SRK	89.1
FR	96.9

Table 10-6 Core recovery by deposit area[4]

Deposit / Recovery %	
Main	94.5
North West	93.8
SE Dome	96.9

10.4 Collar surveys

Prior to 2010, drill hole collar positions were surveyed using total stations. Since then, the KMP mine survey department has used Differential Global Position Systems (DGPS) for this purpose. The DGPS has an accuracy of 0.02 m in northing and easting, and 0.05 m in elevation.

Both UTM (ARC50_35S) and Local Mine Grid coordinate systems were calibrated for the mine license area. Both coordinate systems are accurate, with no risk of spatial distortions, and therefore no risk to the Mineral

[3] Soil = Combined Residual Soil and Soil; length weighted %.
[4] Grouped by Deposit boundary (MP-ALL-PIT-bdry.str, NW-ALL-PIT-bdry.str, SE-ALL-PIT-bdry.str) & length weighted

Resource estimates. Drilling at the South East deposit, and areas more than 1 km from the licence area, were originally recorded in the UTM coordinate system but have since been converted to the Local Grid calibration. DGPS continues to be used for drill hole collar surveys.

Since 2016, all DD and RC holes have been recorded in the Mine Grid system, and all ARC50_35S holes have been transformed into Mine Grid using a seven-point transformation in ArcGIS (Table 10-7).

Table 10-8 lists the transformation parameters used for converting UTM coordinates to the Local Mine Grid coordinates. Both coordinate systems are stored in the database. A Surpac macro was used to calculate the translation and rotation before importing data into the drill database.

Table 10-7 Diamond drill hole collar survey system summary

Year	Orig_Grid_ID
1980	Mine Grid (MG)
1998	ARC50_35S
1998-2008	Mine Grid (MG)
2009	ARC50_35S
2009	Mine Grid (MG)
2010	ARC50_35S
2010	Mine Grid (MG)
2011	ARC50_35S
2011	Mine Grid (MG)
2012	ARC50_35S
2012	Mine Grid (MG)
2013	ARC50_35S
2013	Mine Grid (MG)
2014	ARC50_35S
2014	Mine Grid (MG)
2015	ARC50_35S
2016-2024	Mine Grid (MG)

Table 10-8 Transformation parameters for Kansanshi Mine Grid and UTM Zambia

Type	Horizontal plane		Vertical inclined plane
Origin east	437,403.315 m	Origin east	3,587.183 m
Origin North	8,663,514.836 m	Origin North	13,116.47 m
Translation east	-433,159.514 m	Slope east	87.301 ppm
Translation north	-8,651,380.982 m	Slope north	30.518 ppm
Rotation	-100 01’33.939924”	Constant	4.582
Scale factor	1.00068484		

10.5 Down hole surveys

The downhole spatial position of DD holes was surveyed using a Reflex EZ-TRAC 1.5™ tool, with measurements taken every 50 m, starting at 25 m, and at the end of each hole. Since 2016, the DD survey procedure has been enhanced to ensure greater accuracy in the spatial position of drill hole samples. The updated procedure includes a single-shot survey every 9 m during drill advancement. Once a hole is completed, an additional multishot survey is conducted from the bottom of the hole upwards, also at 9 m intervals. This procedure allows for proactive corrections during drilling and provides a quality control measure upon hole completion.

Before 2013, RC grade control holes were not surveyed downhole due to their shallow depth and low risk of deviation. However, since 2013, RC holes have been routinely surveyed to improve accuracy of defined geological features, such as veins. Since 2020, approximately 87% of RC holes have downhole surveys, with more than 97% of these surveyed RC holes reporting measured parameters within the tolerance levels of 2% for true dip and 5% for true azimuth. Table 10-9 provides a summary of downhole survey compliance between 2015 and 2024, with surveys corrected for magnetic declination as shown in Table 10-10.

Historically, survey measurements were conducted by the drill contractor and recorded at the drill rig, after which the supervising geologist would verify the measured surveys. In 2016, the procedure was upgraded to a web-based system (IMDEXHUB-IQ™), which automates drill hole survey data management with a reporting interface. IMDEXHUB-IQ™ enables wireless transfer from the downhole tools directly to a cloud-based data repository, reducing manual data handling errors and allowing for rapid validation and reporting of results. In 2018, the frequency of RC holes with down-hole survey improved to the current rate of approximately 90%.

Table 10-9 RC drilling down-hole survey compliance

RC Downhole Survey Compliance Summary 2015-2024										
Year	**2015**	**2016**	**2017**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**
Holes drilled	5,940	7,623	5,657	5,358	4,010	3,438	4,362	4,646	6,851	708
Holes DH surveyed	1,558	1015	673	3,437	2,997	2,802	3,962	3,931	5,985	658
Holes DH surveyed %	**26%**	**13%**	**12%**	**64%**	**75%**	**82%**	**91%**	**85%**	**87%**	**93%**
Holes achieving dip compliance	1507	975	659	2,949	2,805	2,767	3,960	3,913	5,937	658
Dip Compliance %	**97%**	**96%**	**98%**	**86%**	**94%**	**99%**	**100%**	**100%**	**99%**	**100%**
Holes achieving azimuth compliance	1542	959	652	3,292	2,934	2,743	3,949	3,925	5,959	656
Azimuth Compliance %	**99%**	**94%**	**97%**	**96%**	**98%**	**98%**	**100%**	**100%**	**100%**	**100%**

Table 10-10 Magnetic declination correction

Magnetic North	**National**	**Local**
Recorded at rig	= Mag north – 2.7°	= Mag north - 12.7°

10.6 Core orientation

Since 2012, DD core orientation has been a standard practice. Various orientation methods have been used over the years, but current programmes employ the Reflex ACT II RD tool, which ensures consistently accurate orientation.

Three metre run lengths are oriented to provide high coverage. Orientation is performed during drilling and core retrieval. Structural data obtained from the Reflex ACT II RD tool is directly uploaded via a cloud server to reduce transcription errors and ensure quality data. This data is managed through IMDEXHUB-IQ, hosted by *iqreflexhub.com*. As an additional check, orientation data is compared with single and multi-shot survey measurements.

Structural alpha, beta, and gamma angle measurements were taken using a kenometer or protractor. These angles were validated by converting them into dip and dip direction format using a Core Solutions (Scott *et al*; 2005) MS Excel-based programme. This method is used to determine the real-space orientations of planar and linear geological fabrics in axially-oriented core. The final conversion of the structural measurements was completed in the Maxwell Datashed SQL drill hole database by applying a conversion calculation.

ITEM 11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Chip and core logging

At the Kansanshi mine site core shed facilities (Figure 11-1), RC chips and diamond core are logged by a team of qualified geologists. Diamond drilling includes infill, extension and sterilisation drilling, with RC drilling focused on operational ore control practices at a closed-grid, accounting for the majority of additional data added over the report period.

Washed RC chips are stored in labelled chip trays and logged for each 3 m chip tray interval. Key lithology and stratigraphic types are the focus of this logging. For holes drilled to intersect veins and narrow geological features (1-3 metre wide), sample intervals are reduced to 1 metre. In these cases, each 1 m drilled interval is sampled and chips are stored in separate tray slots to increase resolution. This enhancement was introduced in August 2022 to improve vein definition.

RC chip logging is conducted using tablet computers loaded with Maxgeo's LogChief software. Data is captured using standardised Log Chief templates and saved directly into a secure MS SQL database. Templates and pre-determined database library codes are pre-set in the LogChief software and accessed via a series of dropdown menus during logging. Any updates and modifications to the templates and codes are agreed upon and then implemented by the database administrator. This methodology reduces transcription errors and ensures consistent and accurate information recording.

Diamond core logging is more extensive than RC chip logging, requiring the recording of lithology, mineralisation, alteration, weathering and structure. Core logging and sampling intervals range from 0.5 m to 3 m and are recorded to two decimal places, with each geology change noted as a separate interval. During logging, data is captured using standard templates (protected Microsoft Excel sheets), which are imported into the MS SQL database via DataShed, Maxgeo's database management software.

Logging protocols and standards are embedded in the templates, which include validation checks to ensure data capture is accurate and of good quality. Validation checks feature dropdown lists locked to database library codes, limiting the types of data captured.

Data import validation tools are setup in DataShed. Tools include *data constraints, –triggers, primary- and foreign keys*. *Data constraints* specify rules for data entries (e.g. dates not exceeding the current date, depths not exceeding the maximum hole depth, TCu% not exceeding 100%). *Triggers* specify SQL actions executed automatically when a specific event occurs in the database (e.g. exceeding maximum depth or exceeding set deviation thresholds trigger and error and stop the import process). *Primary keys* serve as unique identifiers for rows in the database to prevent data duplication, and *foreign keys* links data tables to libraries (e.g. linking and locking fields to libraries) to prevent incorrect entries. Electronic data loading eliminates manual transcription and translation errors, and the storing of erroneous data, ensuring data integrity.

A logging and sampling peer review process is used for final validation and sign-off after completion of core- and chip logging and sampling. This process, developed and enhanced over time, ensures objectivity, relevance, accuracy and consistency of captured data for import into the database. Reviews involve two or more qualified geologists and are conducted at the core shed. Core and chip trays are laid out for inspection and comparison with captured data. Reviews focus on the accuracy of captured data (lithology, stratigraphy, mineralisation, weathering, alteration, structure and other relevant geological or deposit attributes) and the consistent application of logging and database codes. Errors and inconsistencies are identified, corrections are agreed upon, and logging templates updated by the responsible geologists before data import.

The "Geology and Sampling Manual" (Kansanshi Plc, 2021), outlines all relevant and detailed operational procedures and codes for ensuring quality data during the logging of lithology, stratigraphy, alteration, weathering, mineralisation and geological/ deposit attributes. The documents also details sampling standards and guidelines.

Figure 11-1 Kansanshi Mine main core shed (below) showing logging tables (foreground) and core cutting and sample drying facilities towards the centre and rear of the photograph. Additional core storage facilities were constructed in 2020 (Bottom left and right). Core is securely stored in core boxes and protected against exposure to the elements. All facilities are fenced, and access is controlled.

Figure 11-2, Figure 11-3, Figure 11-4, Figure 11-7, Figure 11-8 and Figure 11-9 illustrate the process involved in RC chip and diamond core drilling, logging and photography, sampling, sample handling and dispatch, and analysis activities.

These workflows, while general and generic, have been developed to accommodate and include method-specific requirements for Kansanshi Mine. Technological and software advances are regularly evaluated for their potential to enhance method accuracy, precision, and efficiency. When implemented, these advancements necessitate workflow adjustments.

A recent example is the adoption of Seequent's Imago platform for RC chip and diamond core photography and cataloguing. This platform ensures secure storage of photographs, facilitates easy access to data, and enables interrogation, automated processing, and analysis of drill hole data and attributes. Following its practical implementation, personnel were trained, and workflows were updated accordingly (Figure 11-2 and Figure 11-3).

Figure 11-2 RC Drilling, Logging and Sampling Workflow

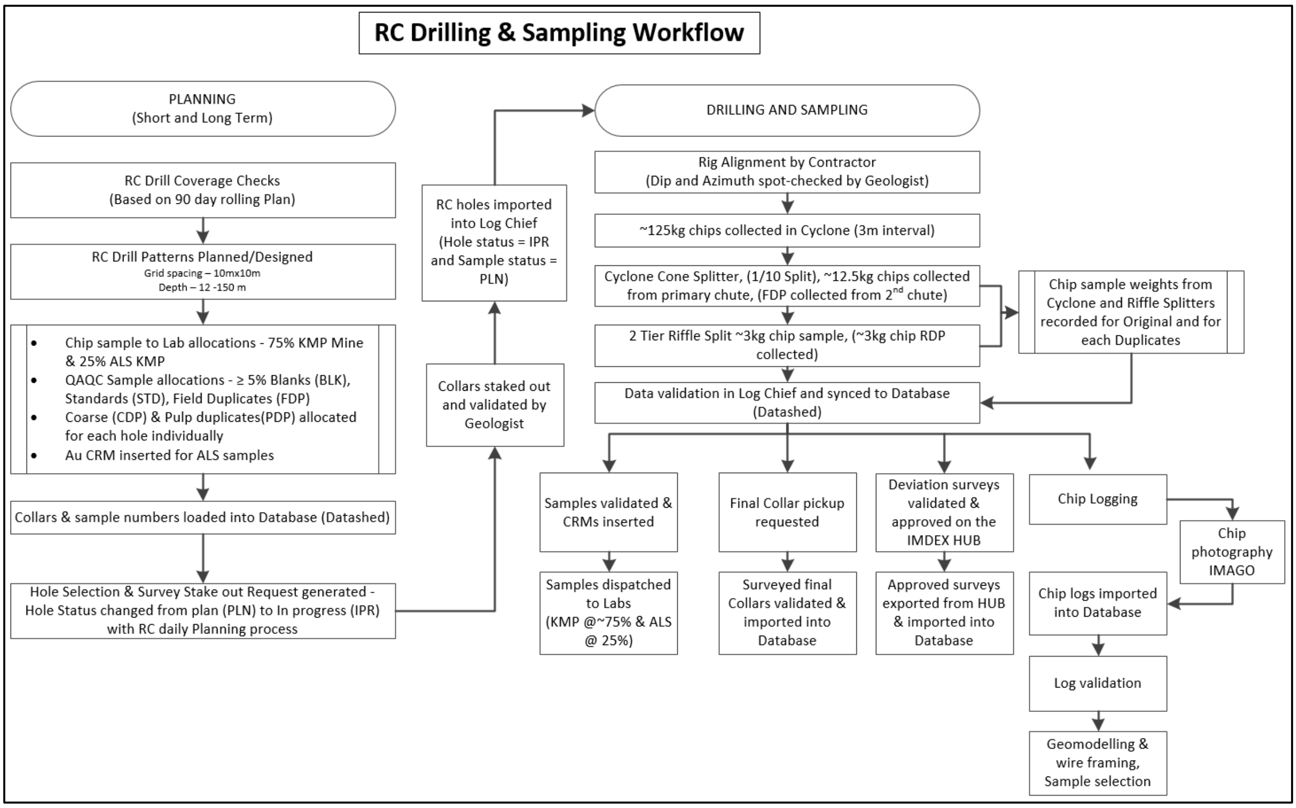
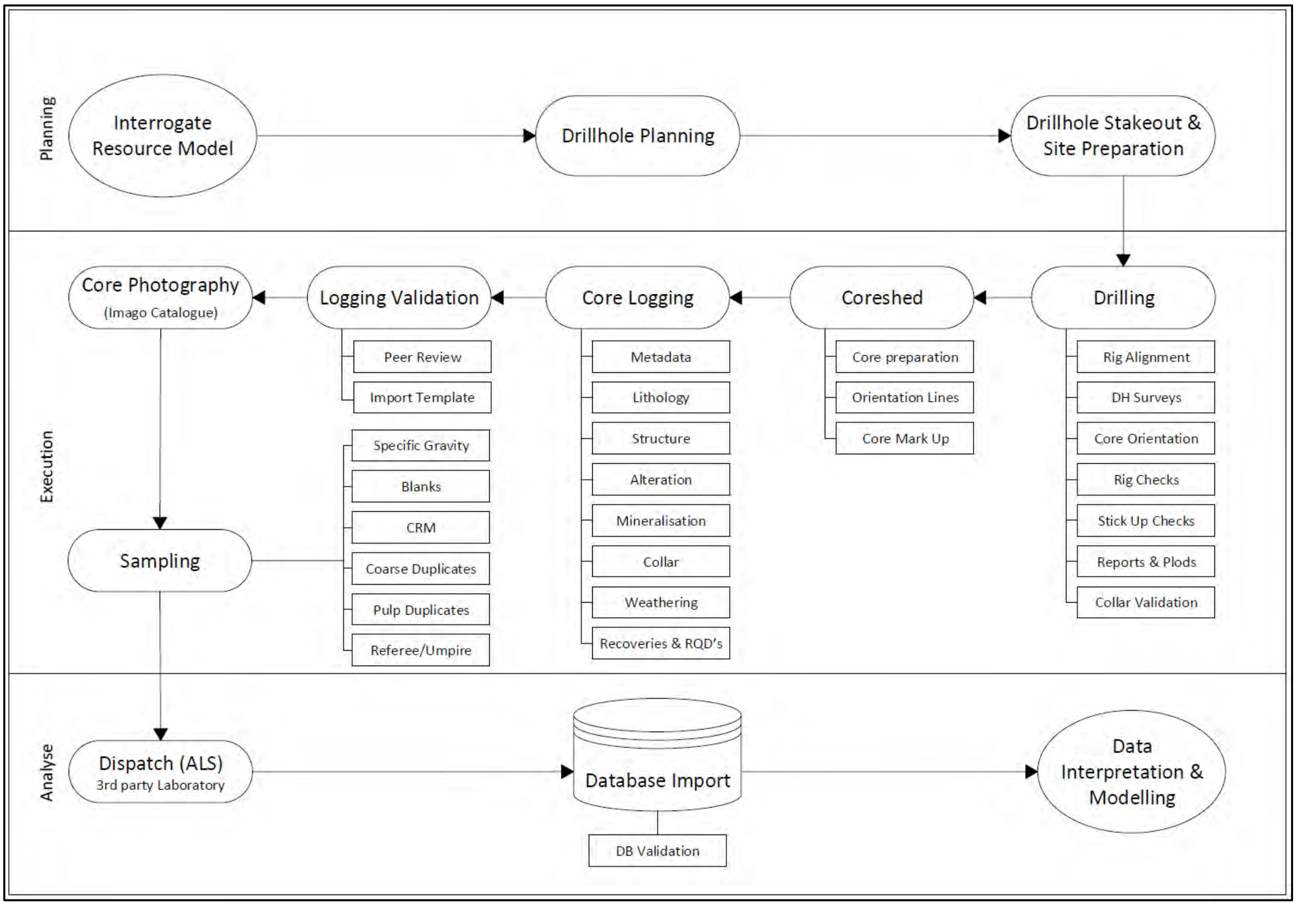


Figure 11-3 DD Drilling, Logging and Sampling Workflow



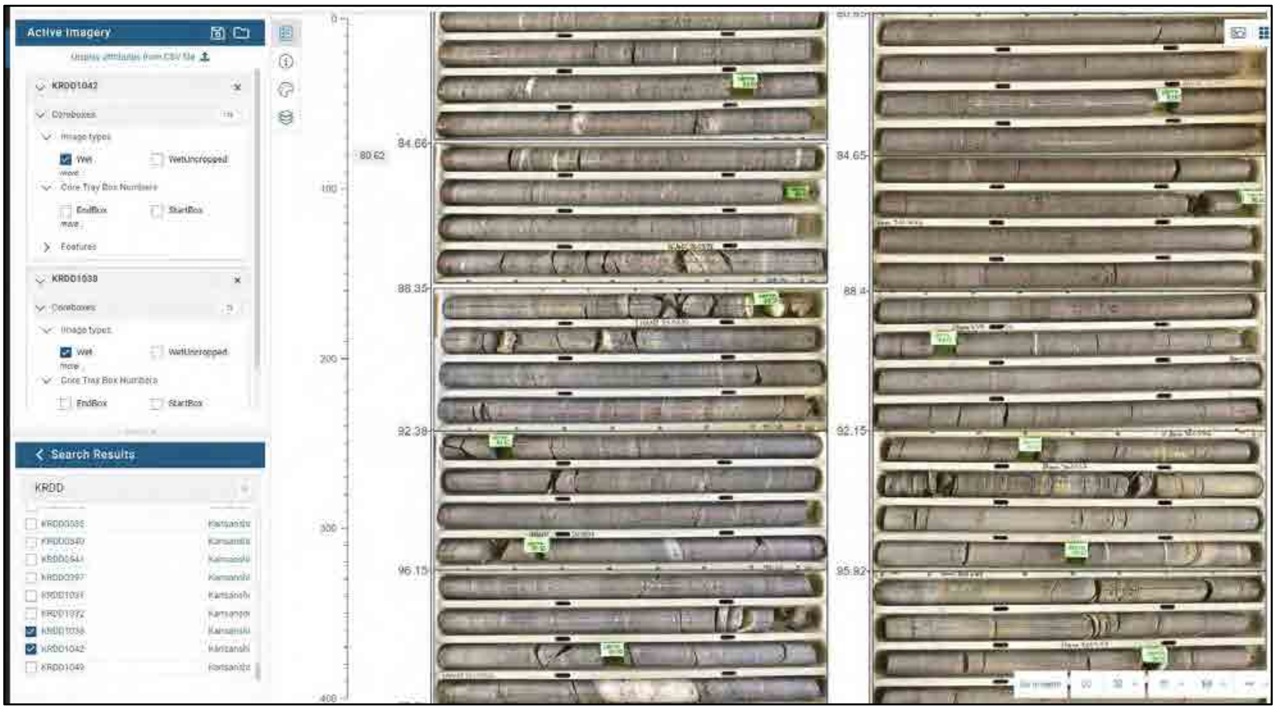
11.2 Chip and core photography and cataloguing

Historically, only uncut core was photographed wet in core trays, with images including a board detailing hole identity, tray number, and core interval. These images were securely saved in a dedicate folder structure on the Kansanshi network servers. Starting from May 2021 for diamond core and July 2022 for RC chips, photographs were taken, catalogued and securely stored on Seequent's Imago cloud-based platform. Figure 11-4 shows two dedicated equipment setups for RC chip (left) and diamond core (right) photography. Photographs are directly saved to Seequent's secure cloud-hosted servers via the Imago desktop interface. This standardised methodology ensures consistent, high-quality, secure images that are safely stored for easy access and processing (Figure 11-5 and Figure 11-6). Historical diamond core photographs have also been catalogued in Imago.

Figure 11-4 Kansanshi RC drill chip (left) and drill core (right) photography setups. Images are recorded and catalogued in Imago, Seequent's cloud-based image cataloguing and processing platform

Figure 11-5 Kansanshi RC drill chip photographs accessed via the Imago's cloud-based catalogue

Figure 11-6 Kansanshi Diamond core photographs accessed via the Imago's cloud-based catalogue



11.3 RC chip and core sampling

All Diamond core samples are analysed by accredited, third-party laboratories, with no core samples being analysed at the Kansanshi Mine Laboratory (KMP_KAN). RC chip samples, on the other hand, are analysed at two on-site laboratories: Kansanshi Mine Laboratory (KMP_KAN) and ALS Kansanshi Laboratory (ALS_KAN). RC chip samples are allocated at a rate of 75% to KMP_KAN Laboratory (unaccredited) and 25% to ALS_KAN Laboratory (accredited). To avoid potential data clustering or bias, drill hole chip samples for submission to ALS_KAN are selected at a rate of 1 in 4 from each drill pattern, ensuring spatially representative data. All chip samples from each individual hole selected were submitted and analysed at ALS_KAN. Both new and historical RC sample data were used for estimates as they remain relevant.

RC chip samples are collected at the rig using a levelled on-rig cone splitter (1/10 split). Two homogenised ~12 kg samples representing each percussed 3 m interval are collected from two separate cyclone chutes. One sample serves as the original, and the second as a field duplicate (FDP) to test method and cyclone cone slitter precision. Modifications to the cyclone and the introduction of a second sample were implemented in 2020 and are now routine QAQC practice. When drilling is planned to intersect veins, intervals are reduced to 1m for increased definition, and two ~ 4kg samples are collected from the cyclone chutes.

The RC field samples are subsequently split with a Jones riffle splitter down to ~3 kg. Samples are bagged in numbered calico bags and delivered to the Kansanshi core and chip sample storage facility before sorting and inserting the QC samples (CRMs and blanks). Samples are later dispatched to the respective laboratories. At KMP_KAN RC chip samples are analysed for copper (TCu) and acid soluble copper (ASCu). At ALS_KAN chips samples are analysed for copper (TCu), acid soluble copper (ASCu), gold (Au) and nickel (Ni).

After logging, diamond core is marked out for sampling from top to bottom of each hole. Sample lengths range from 0.5 metres to 3.0 metres according to mineralisation and geological boundaries. High-grade and native copper intervals (greater than 3% Cu) are flagged for quality control (QC). QAQC samples, including Certified Reference Materials (CRM's), blanks and duplicates are routinely inserted as per rates defined in KMP's standard protocols. Umpire sample analyses are also used to confirm confidence and ensure acceptable accuracy. All sampled drill core is analysed for total copper, acid soluble copper and gold. Marked drill core is cut with a diamond saw; one half is submitted to the laboratory for analysis, and the remaining half is retained in labelled core trays. Core trays are stored safely and securely on site at the dedicated core storage facilities (Figure 11-1).

Figure 11-7 Kansanshi Laboratory (KMP_KAN) RC chip sampling and analysis process flow

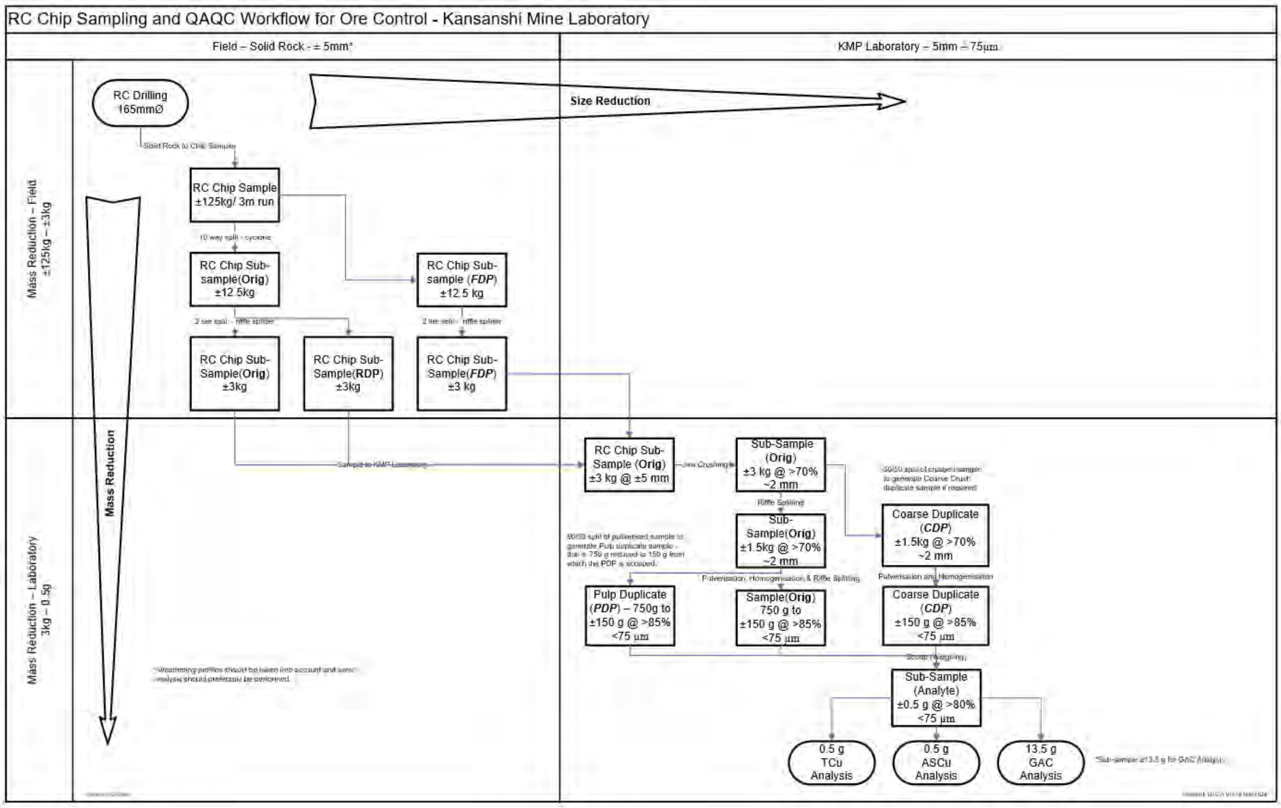
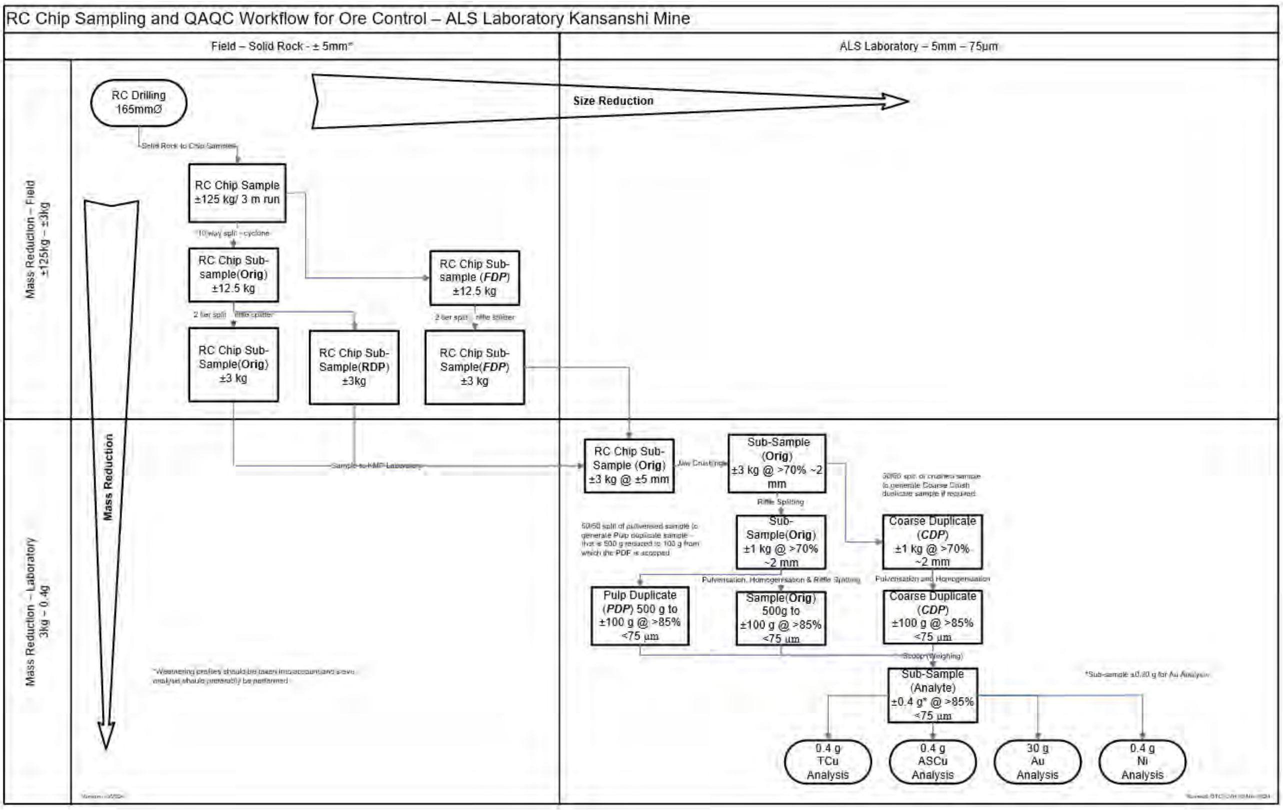


Figure 11-8 ALS Kansanshi Laboratory RC chip sampling and analysis process flow



11.4 Sample logistics, dispatch and custody

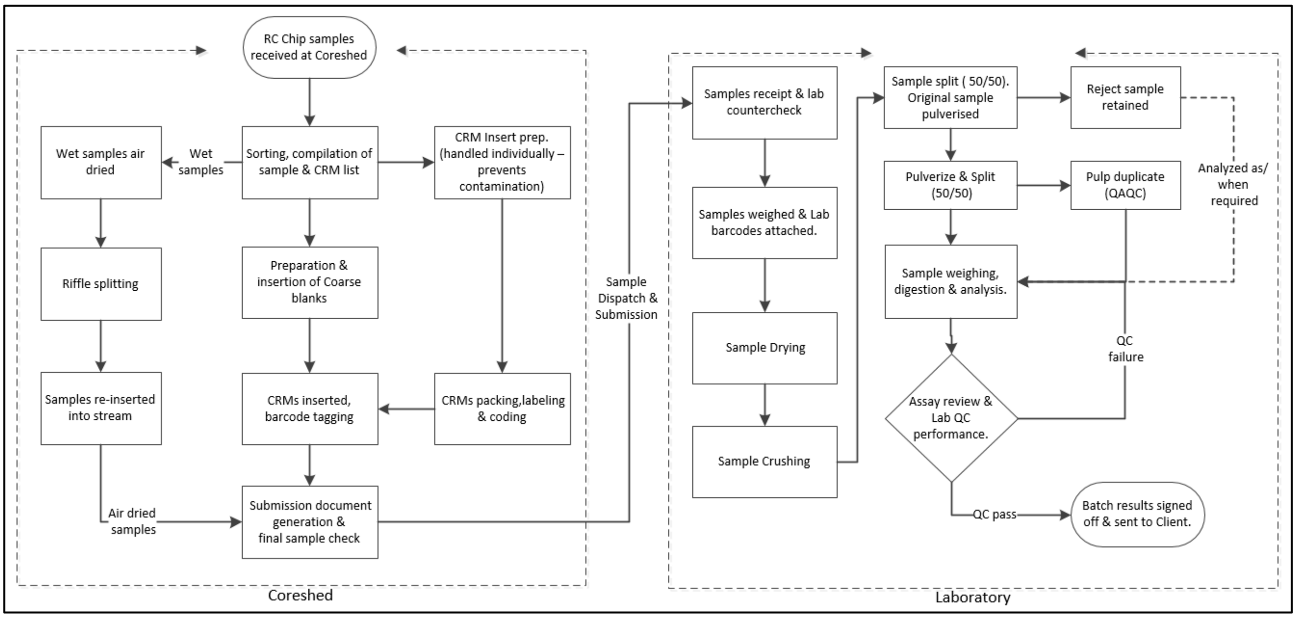
RC Chip samples are collected and transported from the respective in-pit drill sites to the core shed for sorting. Wet bulk samples are placed in sample trays for air drying for approximately 2 to 4 hours. Dry samples are riffle split and re-inserted into the sample stream in sequence. CRMs are individually packed, labelled, coded and inserted, followed by barcoding. Sample submission documents are generated, and a final sample check is conducted before dispatching the samples to KMP_KAN and ALS_KAN laboratories.

At the laboratories, samples are cross-checked against submission documents prior to weighing and laboratory barcoding. Chip samples are prepared for drying in ovens at 105°C. After drying, samples undergo crushing and pulverising, followed by weighting of the pulp samples before digestion. Analytes are then measured using atomic absorption spectrographic analysis (AAS finish).

Assay results are reviewed for QC performance. Batches with satisfactory results are signed off, and final results forwarded to the Kansanshi Geology Team. In cases of QC failures, the affected sample batches are re-run.

Diamond core logistics and handling follow a similar process, with the main distinction being that samples are exclusively submitted to ALS_KAN for analysis.

Figure 11-9 RC chip sample logistics, administration and preparation workflow.



11.5 Sample preparation

RC chip and DD core samples were oven dried at temperatures of 80 or 105 °C. Once dry, samples were jaw crushed to a ~2 mm size. Crushed samples were then repeatedly split to obtain sample mass of 1 to 1.5 kg and subsequently pulverised using a ring mill pulveriser with a chrome steel ring set to achieve 85% passing 75 µm. Particle size analyses were conducted to ensure compliance with grinding and pulverisation specifications.

Pulp samples were transferred onto a rubber rolling mat and homogenised by repeatedly rolling the pulp back upon itself. A scoop of 150 - 250 g of the homogenised pulp sample was then placed into its numbered envelope, ready for analysis.

11.6 Quality Assurance and Quality Control (QAQC)

KMP utilized the services of three analytical laboratories: the KMP Mine Laboratory (KMP_KAN), own by FQM and not an independent laboratory, and two external independent laboratories, ALS Kansanshi laboratory (ALS_KAN), run by ALS and SGS Kalulushi (SGS_KAL), run by SGS. Both KMP_KAN and ALS_KAN facilities are located on-site, whereas SGS is located in Kalulushi, Copper Belt Province, Zambia. ALS_KAN laboratory's accreditation was renewed on 23 June 2023 by SADCAS (ISO/ IEC 17025:2017 accreditation "*TEST-5 0027*").

To ensure representative sampling and analyses, Kansanshi has developed robust QAQC programmes and practices designed and executed according to industry standard. These practices ensure that sampling and analytical errors are minimised, identified, and promptly resolved.

QAQC sample insertion rates have varied over time for RC chip and diamond core samples. Insertion rates for CRMs, blanks and duplicates have consistently been maintained above 5%, ensuring comprehensive coverage across all sample types. Insert rates and performance are detailed in the respective sections below. Insert rates per QAQC type are monitored weekly and reported monthly, using DataShed's QAQCR reporting functionality. All QAQC data is stored in the DataShed SQL drill hole database.

Comprehensive monthly laboratory audits are conducted by KMP geologists, followed by formal reporting of laboratory performance to on-site laboratories. This ensures acceptable laboratory performance is maintained and supports robust QAQC practices.

Laboratory audit items include safety and housekeeping, sample preparation, barcoding and tracking in LIMS, drying procedures and temperatures, crushing and splitting, cleanliness to prevent contamination, crush and pulverising size, screen testing, homogenisation, sample and aliquot weights, duplicate procedures and collection, CRM and blank processing and sequencing, reject collection and storage, analysis calibration, consistent batch sizes, digest solution characteristics, equipment maintenance and calibration frequency, adherence to or deviations from laboratory protocols, procedures and processes, laboratory QAQC repeats, CRM and blank insertion rates.

Monthly QAQC review meetings are held between the on-site laboratories and the geology team to review performance and discuss remedial actions in cases of poor performance.

In addition to monthly audits and review meetings, geologists conduct weekly internal QAQC performance reviews to maintain proactive oversight.

11.6.1 Certified Reference Materials (CRMs)

CRMs were inserted to test and ensure analytical accuracy and to assess potential biases in laboratory processes. During the 2019 to 2024 period, 16 different CRMs were sourced from suppliers using materials similar to those found in Kansanshi ore or mines, with comparable copper grade ranges. The CRM materials included oxide, sulphide and waste types. For instance, AMIS0474, prepared by African Minerals Services, contains low-grade sulphide copper and gold mineralisation and has been used for RC chip and diamond core sampling since 2014.

CRM insertion rates have varied over time, ranging from 8% to 9% for RC chip samples and 7% to 10% for diamond core samples. Multiple different CRMs were used for both RC chips and diamond core (Table 11-1 to Table 11-4), ensuring comprehensive coverage across sample batches, material types and grade ranges. Recent insertion rates slightly exceed previously reported rates of 5% to 6%.

11.6.2 Blank samples

Blanks were inserted into the sample sequence to check for contamination during sample preparation and to identify any sample swapping. Three different blanks (BLK) were used, including two certified powder format by AMIS, and one prepared from locally sourced syenite material. This coarse material was crushed to RC chip size and independently analysed by ALS_KAN and KMP_KAN. Laboratory crosschecks confirmed the blank value for the coarse blank. For RC samples, blanks were pre-assigned during the planning phase, while for diamond core, blank samples were inserted systematically between and after high-grade intersections and at the end of the drill hole sample sequence. Analysed values were monitored per batch to prompt remedial action if required. Blank insertion rates varied between 6% and 7% for RC samples and between 5% and 9% for diamond core samples. Table 11-1 to Table 11-4 summarise the weight averaged CRM and blank insertion rates.

Table 11-1 ALS_KAN Diamond Core CRM insertion rates

ALS Diamond Core CRM Insertion rates (weight averaged for 2019-2024)					
CRM Type	# Core Samples	# different CRMs used	# CRMs Inserted	CRMs Insert ratio	CRM Insert %
BLK	23,917	1-2	2,133	1:11	9
CRM	23,917	2-10	2,433	1:10	10
LAB	23,917	8-18	7,489	1:3	31

Table 11-2 SGS Diamond Core CRM insertion rates

SGS Diamond Core CRM Insertion rates (weight averaged for 2019-2024)					
CRM Type	# Core Samples	# different CRMs used	# CRMs Inserted	CRMs Insert ratio	CRM Insert %
BLK	2,076	1-2	111	1:19	5
CRM	2,076	4-6	143	1:15	7
LAB	2,076	3-4	610	1:3	29

Table 11-3 ALS_KAN RC Chip CRM insertion rates

ALS Kansanshi RC Chip CRM Insertion rates (weight averaged for 2019-2024)					
CRM Type	# Chip Samples	# different CRMs used	# CRMs Inserted	CRMs Insert ratio	CRM Insert %
BLK	120,496	2-3	7,633	1:16	6
CRM	120,496	6-9	10,349	1:12	9
LAB	120,496	10-21	45,744	1:3	38

Table 11-4 KMP_KAN RC Chip CRM insertion rates

KMP Kansanshi RC Chip CRM Insertion rates (weight averaged for 2019-2024)					
CRM Type	# Chip Samples	# different CRMs used	# CRMs Inserted	CRMs Insert ratio	CRM Insert %
BLK	301,897	2-3	19,687	1:15	7
CRM	301,897	7-8	25,653	1:12	8

11.6.3 Duplicate samples

Duplicate samples are routinely inserted to assess the precision of sampling and analyses. These include field duplicates (FDP), riffle split duplicates (RDP), coarse crush duplicates (CDP), pulp duplicates (PDP), laboratory pulp duplicates (LDP) and referee or umpire pulp duplicates (PDX). The responsible geologist assigns duplicate samples according to KMP's standard sampling procedure for both diamond core and RC chip samples.

FDPs and RDPs duplicates are collected at source during drilling. FDPs are collected during the first sample mass reduction stage from one of the two sample chutes on the drill rig cyclone. RDPs are collected from the riffle splitters during the second stage of mass reduction. CDPs are collected during the third stage of mass reduction at the laboratory sample preparation facilities, where the coarse duplicate is collected when a ~3kg sample is split down to ~1.5kg.

Sample size reduction begins during crushing and continues with a second reduction during pulverising by laboratory personnel. A pulp duplicate sample is collected after the sample has been pulverised and homogenised, just before analysis. Laboratories insert and track laboratory duplicates as part of their internal precision checks. Umpire pulp duplicates (PDX) are analysed between laboratories to evaluate potential biases. Table 11-5 to Table 11-8 summarise duplicate sample insertion rates since September 2019.

Table 11-5 ALS-KAN Diamond Core duplicate insertion rates

ALS_KAN Diamond Core Duplicate Insertion rates (weight averaged for 2019-2024)				
Duplicate Sample Type	**# Core Samples**	**# Inserted**	**Insert ratio**	**Insert %**
Coarse Crush Duplicate (CDP)	23,917	1,529	1:16	6
Laboratory Duplicate (LDP)	23,917	1,226	1:20	5
Pulp Duplicate (PDP)	23,917	1522	1:16	6

Table 11-6 SGS_KAL Diamond Core duplicate insertion rates

SGS Diamond Core Duplicate Insertion rates (weight averaged for 2019-2024)				
Duplicate Sample Type	**# Core Samples**	**# Inserted**	**Insert ratio**	**Insert %**
Laboratory Duplicate (LDP)	2,076	200	1:10	10
Pulp Umpire (PDX)	2,076	2,076	1:1	100

Table 11-7 ALS_KAN RC Chip duplicate insertion rates

ALS Kansanshi RC Chip Duplicate Insertion rates (weight averaged for 2019-2024)				
Duplicate Sample Type	**# Chip Samples**	**# Inserted**	**Insert ratio**	**Insert %**
Coarse Crush Duplicate (CDP)	120,496	6,321	1:19	5
Field Duplicate (FDP)	120,496	6,389	1:19	5
Laboratory Duplicate (LDP)	120,496	7,757	1:16	6
Pulp Duplicate (PDP)	120,496	6,321	1:19	5
Riffle Duplicate (RDP)	120,496	1,026	1:117	1
Umpire Pulp Duplicate (PDX)	120,496	3,838	1:31	3

Table 11-8 KMP_KAN RC Chip duplicate insertion rates

KMP Kansanshi RC Chip Duplicate Insertion rates (weight averaged for 2019-2024)				
Duplicate Sample Type	**# Chip Samples**	**# Inserted**	**Insert ratio**	**Insert %**
Coarse Crush Duplicate (CDP)	301,897	15,646	1:19	5
Field Duplicate (FDP)	301,897	15,936	1:19	5
Pulp Duplicate (PDP)	301,897	15,645	1:19	5
Riffle Duplicate (RDP)	301,897	2,596	1:117	1
Umpire Pulp Duplicate (PDX)	301,897	2,101	1:145	1

11.7 RC chip sample weight measurement

Sample weights are measured at source to limit sampling error arising from poor practices, faulty equipment, seasonal changes, and geological variations, all of which can influence sample weight and quality. RC cyclone and subsequent riffle split sample weights are measured to ensure consistency of sample mass during sampling and splitting. The mean weight of RC cyclone sample (FDP) is 11.66 kg, while the mean weight of

Jones riffle split sample (RDP) is 2.87kg. These weights align closely with expected chip sample cyclone and riffle split weights.

There is a strong positive correlation between the original samples weights and cyclone duplicate (FDP) weights, with original sample weights displaying a slightly high bias. 99% of the data has a Half Absolute Relative Difference (HARD) of ≤10%, indicating good precision for the FDPs.

Riffle duplicate (RDP) weights are measured in-pit at the sampling station. Jones riffle splitters are used for a 50/50 split of the chip samples. RDPs exhibit a strong positive correlation with the original sample weights, with the original sample weights displaying a slight positive bias. 98% of the RDP data has a HARD of ≤10%, indicating good precision (Figure 11-10 and Figure 11-11).

Figure 11-10 Scatter and HARD plots for RC Field Duplicate (FDP) Cyclone Sample Weights (2019-2024)

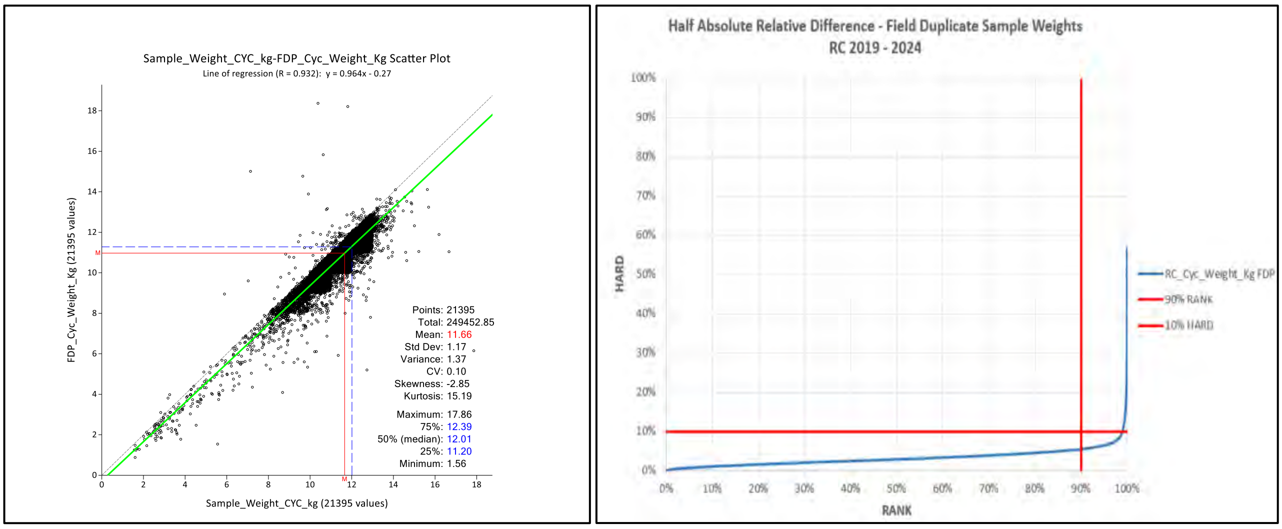


Figure 11-11 QQ Plot and Histogram for RC Field Duplicate (FDP) Cyclone Sample Weights (2019-2024)

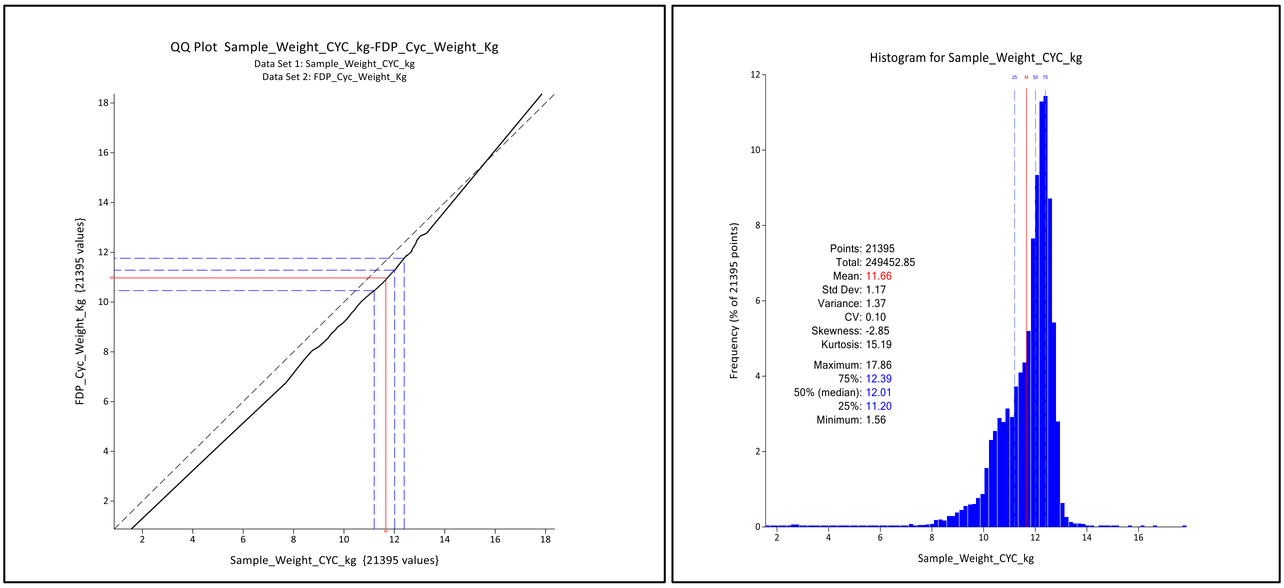


Figure 11-12 Scatter and HARD plots for RC Riffle Duplicate (RDP) Sample Weights (2019-2024)

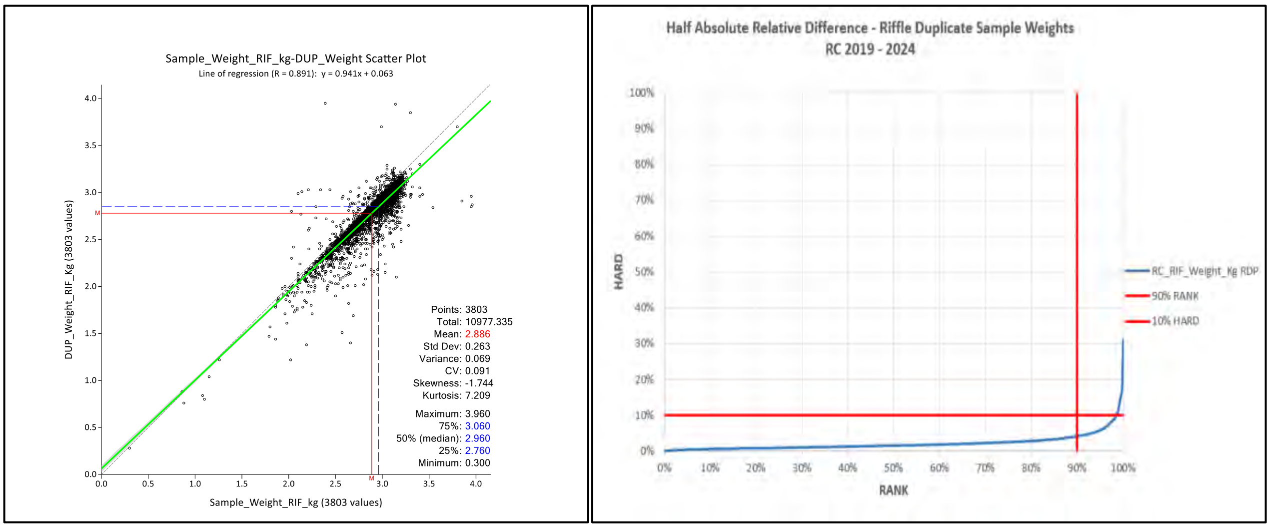
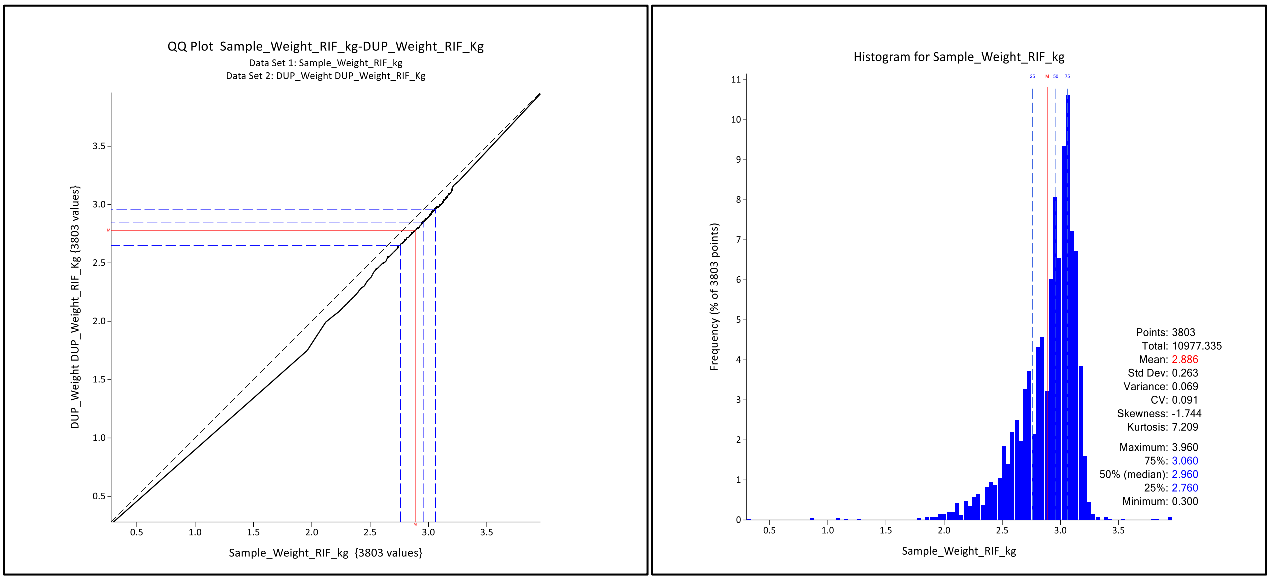


Figure 11-13 Scatter Plot and Histogram for RC Riffle Duplicate (RDP) Sample Weights (2019-2024)



11.8 Laboratory analytical methods

Diamond core and RC chip sample analyses were conducted at ALS Kansanshi (ALS_KAN) and Kansanshi Mine Laboratory (KMP_KAN). Historically, some diamond samples were analysed at ALS Chemex Johannesburg and Genalysis Johannesburg, both accredited laboratories. Table 11-9 provides details on the number of samples and respective laboratories used between September 2019 and February 2024.

Table 11-9 Kansanshi DD and RC sample summary by Laboratory (2019-2024)

DD Holes DSKansanshiRS			
Laboratory	**Accredited**	**Samples**	**Year**
ALS Kansanshi - ALS_KAN	Yes	2,021	2019
		11,506	2020
		9,115	2021
		8,830	2022
		62	2023
SGS Kalulushi - SGS_KAL	Yes	1,936	2020
		254	2021
		140	2022
*FQM submitted samples only (no Lab Checks included)			

RC Holes DSKansanshiGC			
Laboratory	**Accredited**	**Count_Samples**	**Year_Analysis**
ALS Kansanshi - ALS_KAN	Yes	8,317	2019
		30,587	2020
		42,760	2021
		34,545	2022
		43,222	2023
		2,942	2024
Kansanshi Mine Laboratory - KMP_KAN	No	13,380	2019
		84,421	2020
		104,607	2021
		82,903	2022
		100,809	2023
		13,041	2024
*FQM submitted Samples only (no Lab Checks included), Holes from South East, North West and Main deposits only - no stockpile data			

Diamond core sample analysis included total copper (TCu), acid soluble copper (ASCu), cyanide leach copper (CnCu) and gold (Au). CnCu analysis was limited to diamond drilling from 2011 to 2014 (<68% of total diamond samples). Approximately 16% of diamond samples were also analysed for multiple elements[5].

RC samples at KMP_KAN were analysed for TCu, ASCu and gangue acid consumption (GAC). Meanwhile, 25% of RC samples were sent to ALS_KAN for TCu, single acid soluble copper (SSCu), and Au analysis.

Diamond pulp rejects were stored at ALS_KAN and RC pulp rejects were stored at the KMP core shed facility.

Below is a brief description of the diamond core sample analysis conducted by ALS_KAN:

- Total copper (TCu) was analysed using a four-acid digest and an Inductively Coupled Plasma with Atomic Emission Spectrometry (ICP-AES) finish.

[5] 4A_ICPXS & ME-MS61r/ME-MS61 methods employed in 2009, 2012-2015.

- Acid soluble copper (ASCu) was analysed using a single sulphuric acid leach and an Atomic Absorption spectrometry (AAS) finish.
- Cyanide soluble copper (CnCu) was analysed using a cyanide leach and an AAS finish (~33% of drill holes, from 2011-2015).
- Gold (Au) was analysed using a fused precious metal bead, which was then digested in acid and analysed with AAS against a matrix-matched standard.

RC chip samples were analysed at KMP_KAN (75%) and ALS_KAN (25%). The analysis included:

- Total copper (TCu) digestion with a four-acid solution and AAS finish.
- Acid soluble copper (ASCu) digestion with a single acid leach and an AAS finish.
- Gangue acid consumption (GAC) measurement using standard volume titration techniques.
- Additionally, 25% of the RC samples submitted to ALS_KAN were analysed for Au using a fused precious metal bead, digested in acid and analysed with AAS against a matrix-matched standard.

11.9 QAQC analysis and results

QAQC analysis of RC chip and diamond core samples is comprehensive, with acceptable insertion rates for CRMs, blanks, field duplicates, riffle duplicates, coarse duplicates, pulp duplicates and umpire duplicates. Defined procedures, regular monthly laboratory audits, and a dedicated QAQC database, combined with analysis and standardised monthly reporting and review enable proactive management of QAQC sample results. Kansanshi Mine employs certified geologists, well-trained permanent geological staff, and a dedicated database administrator to ensure that QAQC standards and procedures are supported.

Failed batches are resubmitted for analysis to ensure that representative samples are used for estimating block model grades. Ongoing reviews and audits of practices and results, along routine umpire checks between the two on-site and external laboratories, are part of the continuous improvement process.

11.9.1 Umpire sample analysis results

Umpire samples were selected from a range of sample grades in order to validate the accuracy of primary laboratory analytical results.

Initial RC umpire analyses were completed in 2013 (LDX series). Further comparative studies between KMP_KAN and ALS_KAN analyses were conducted during 2018 into 2019, revealing strong positive correlations. During this period, duplicate analyses demonstrated good precision, with approximately 90% of samples pairs showing HARD values within ≤10%. These results indicated that the primary laboratories achieved acceptable analytical accuracy and precision.

As part of routine QAQC practices, the submission of RC umpire samples and subsequent analyses of performance continued from 2019 to 2024. Recent results have shown a continuation of acceptable laboratory performance at both KMP_KAN and ALS_KAN. Strong positive correlations and HARD values (≤10%) of 94% and 92% respectively, also indicate improvement (Figure 11-14 and Figure 11-15).

Figure 11-14 Scatter and HARD plots of RC umpire (PDX) datasets for TCu (2019-2024); ALS_KAN pulps umpired by KMP_KAN

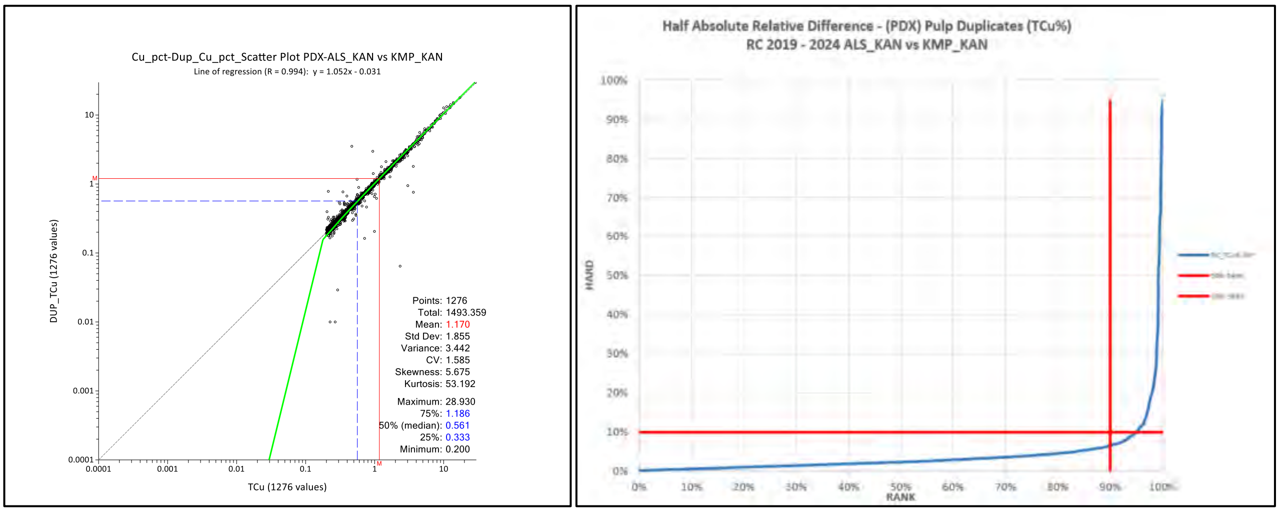


Figure 11-15 Scatter and HARD plots of RC umpire (PDX) datasets for TCu (2019-2024); ALS_KAN pulps umpired by KMP_KAN

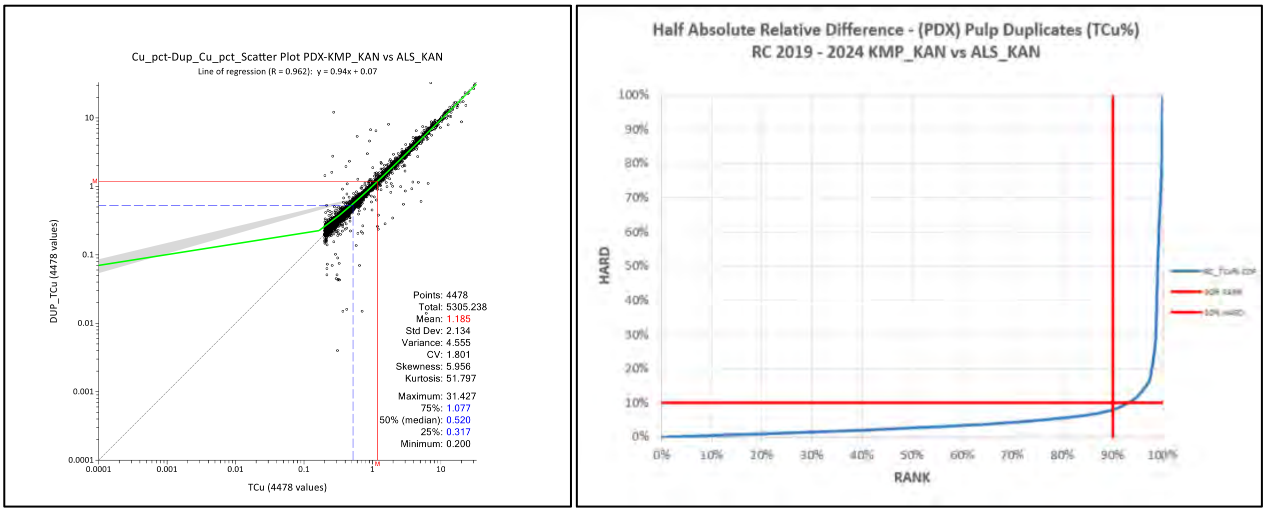
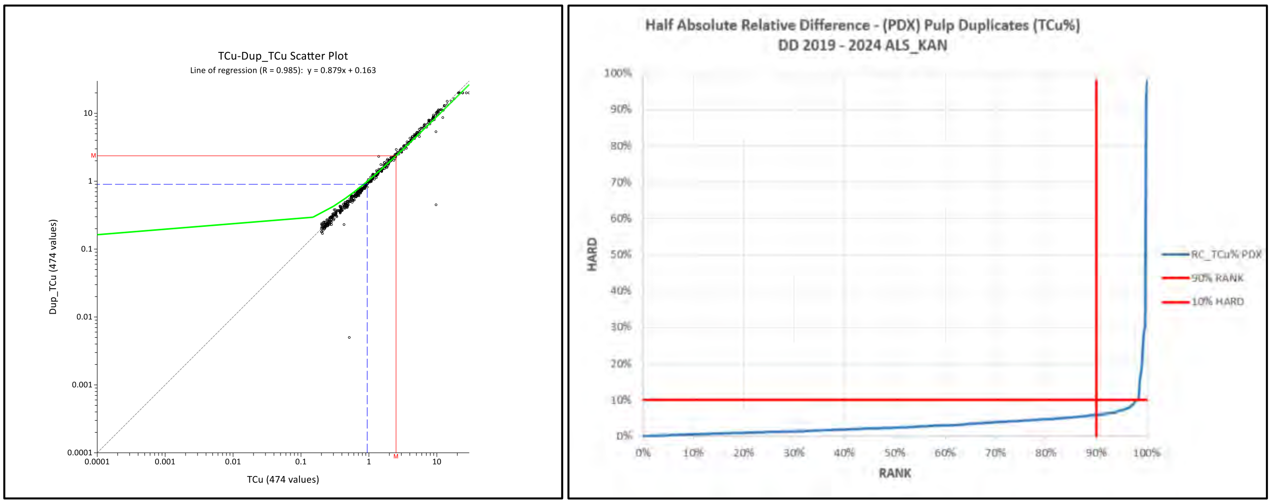


Figure 11-16 Scatter and HARD plots of DD umpire (PDX) datasets for TCu (2019-2024); ALS_KAN pulps umpired by SGS_KAL



Before 2015, umpire results indicated a risk of sample number swapping. Subsequent improvements in sampling standards, laboratory protocols, and ALS_KAN management, along with lab accreditation, have mitigated this risk. Between 2019 and 2024, diamond umpire samples were submitted to SGS Kalulushi (SGS_KAL), an accredited third-party laboratory. Diamond umpire analyses demonstrated a strong positive correlation, with 97% of sample pairs within a HARD value of ≤ 10% (Figure 11-16). These results indicate acceptable analytical accuracy and precision.

11.9.2 CRM sample analysis and results

The averaged CRM failure rate between 2019 and 2024 amounted to approximately 5.0%. Analytical bias was monitored through the performance of CRMs, which reported to be within acceptable limits for TCu, ASCu and Au. Control charts (Figure 11-17 to Figure 11-20) illustrate the performance of multiple CRMs for diamond drill core samples by instances over time.

Figure 11-17 DD control chart for AMIS 0365 – TCu by instance - four acid digest with AAS finish (2019-2024)

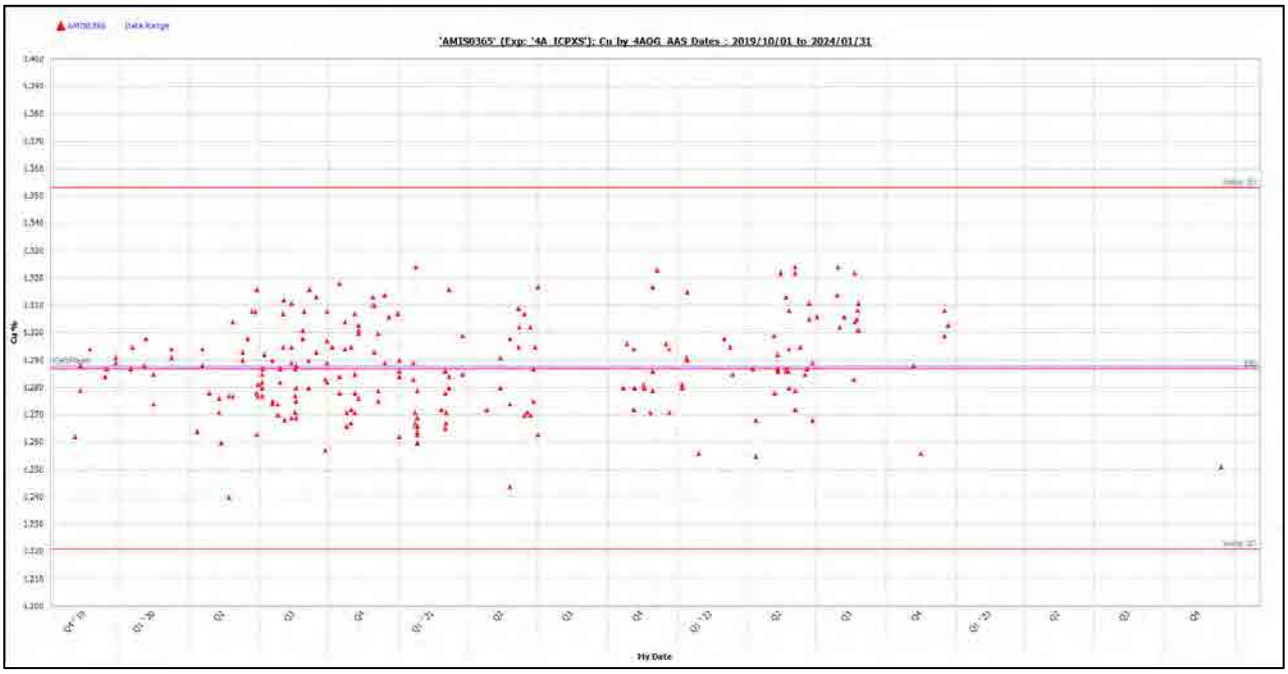


Figure 11-18 DD control chart for AMIS 0370 – TCu by instance - four acid digest with AAS finish (2019-2024)

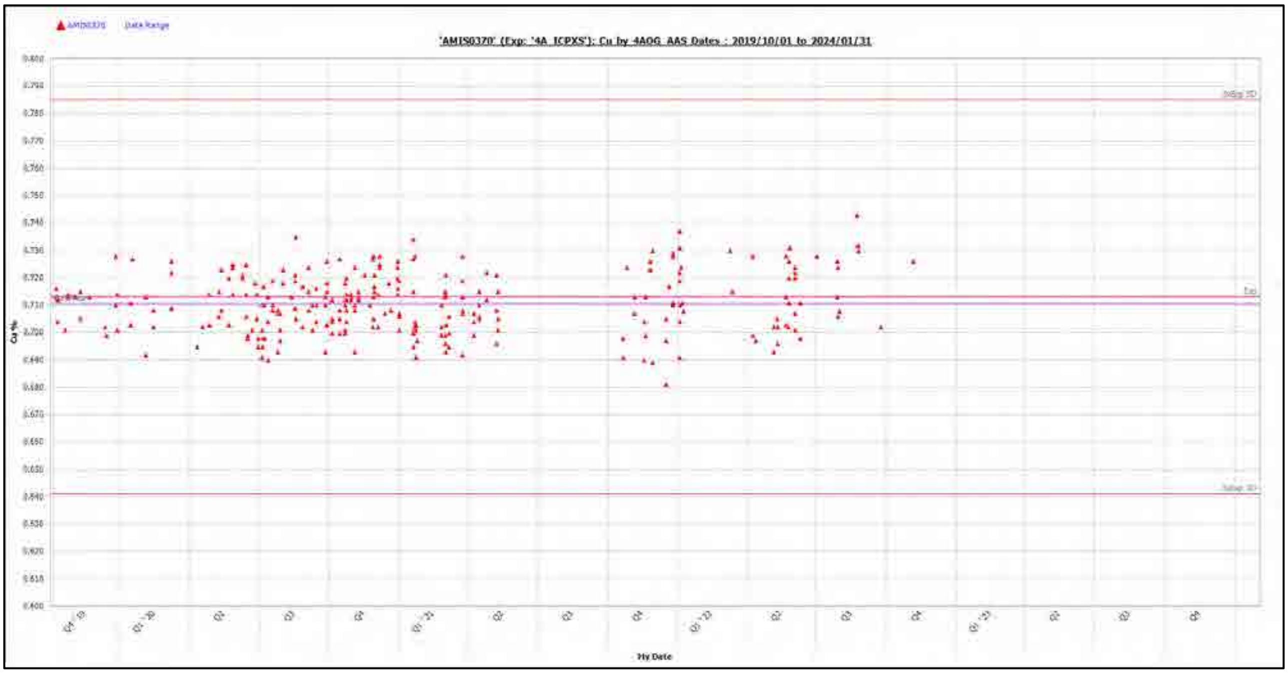


Figure 11-19 DD control chart for AMIS 0474 – TCu by instance - four acid digest with AAS finish (2019-2024)

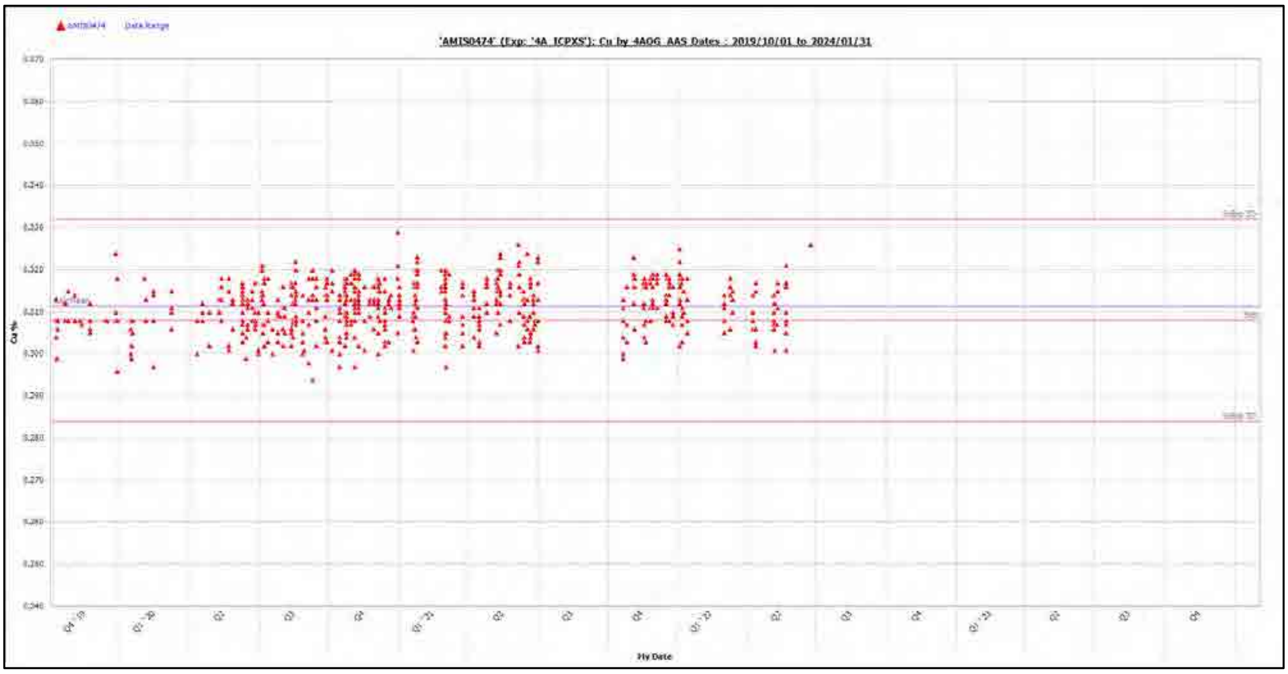
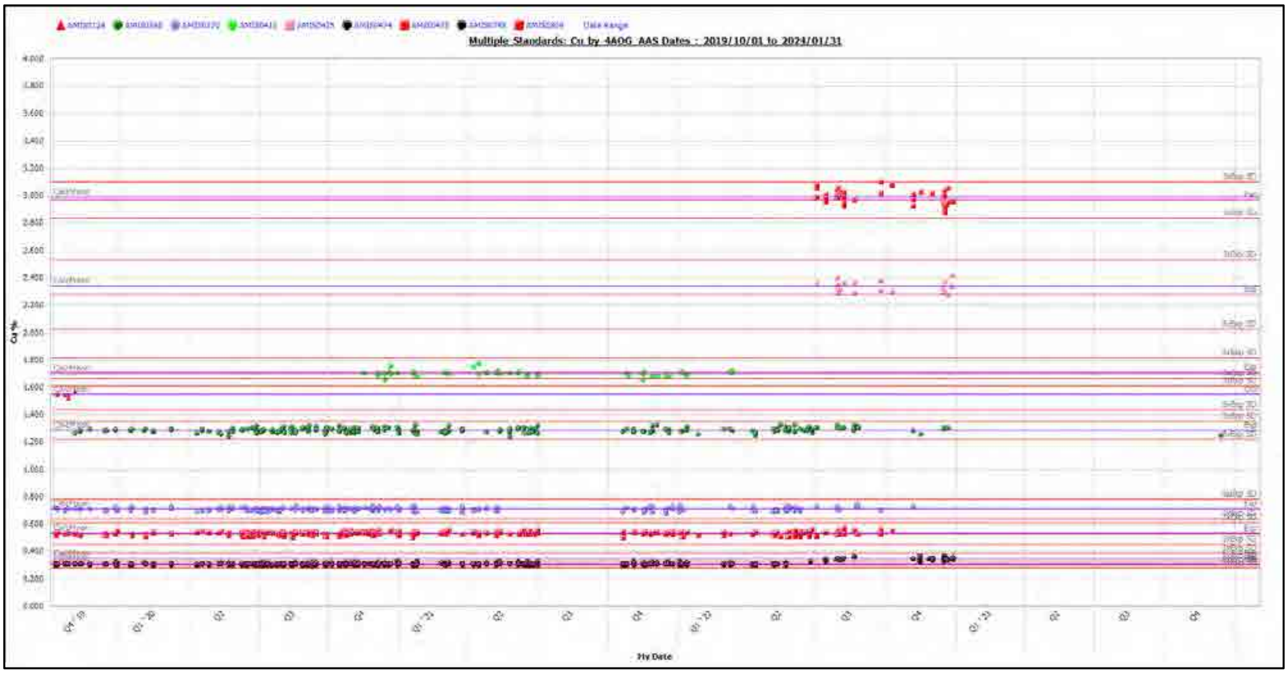


Figure 11-20 DD control chart for multiple CRMs – TCu by instance - four acid digest with AAS finish (2019-2024)



CRM failure rates for RC samples between 2019 and 2024 were 1.0% for ALS_KAN and 2.0% for KMP_KAN. Analytical bias was monitored through CRM performance and reported to be within acceptable limits for TCu, ASCu and Au, showing improvement compared to previous reporting. Figure 11-21 and Figure 11-25 are control charts illustrating the performance of multiple RC and diamond core CRMs.

Figure 11-21 RC control chart for AMIS 0365 – TCu by date - four acid digest with AAS finish (2019-2024)

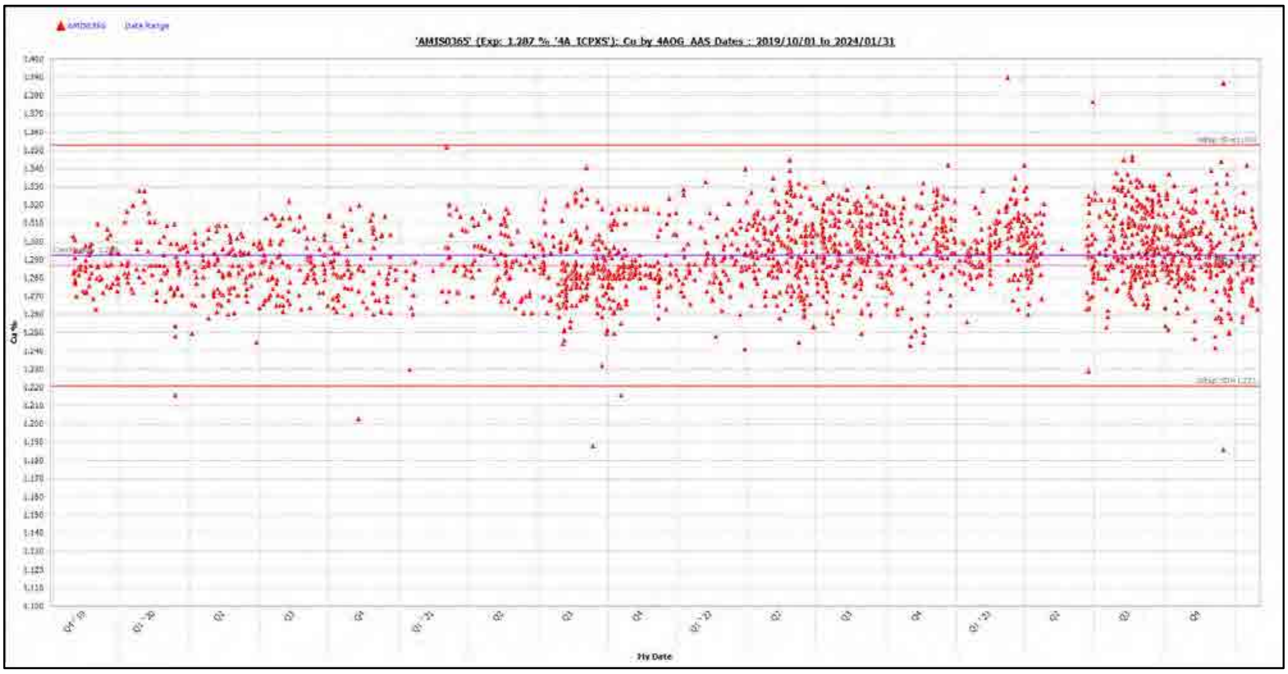


Figure 11-22 RC control chart for AMIS 0370 – TCu by instance - four acid digest with AAS finish (2019-2024)

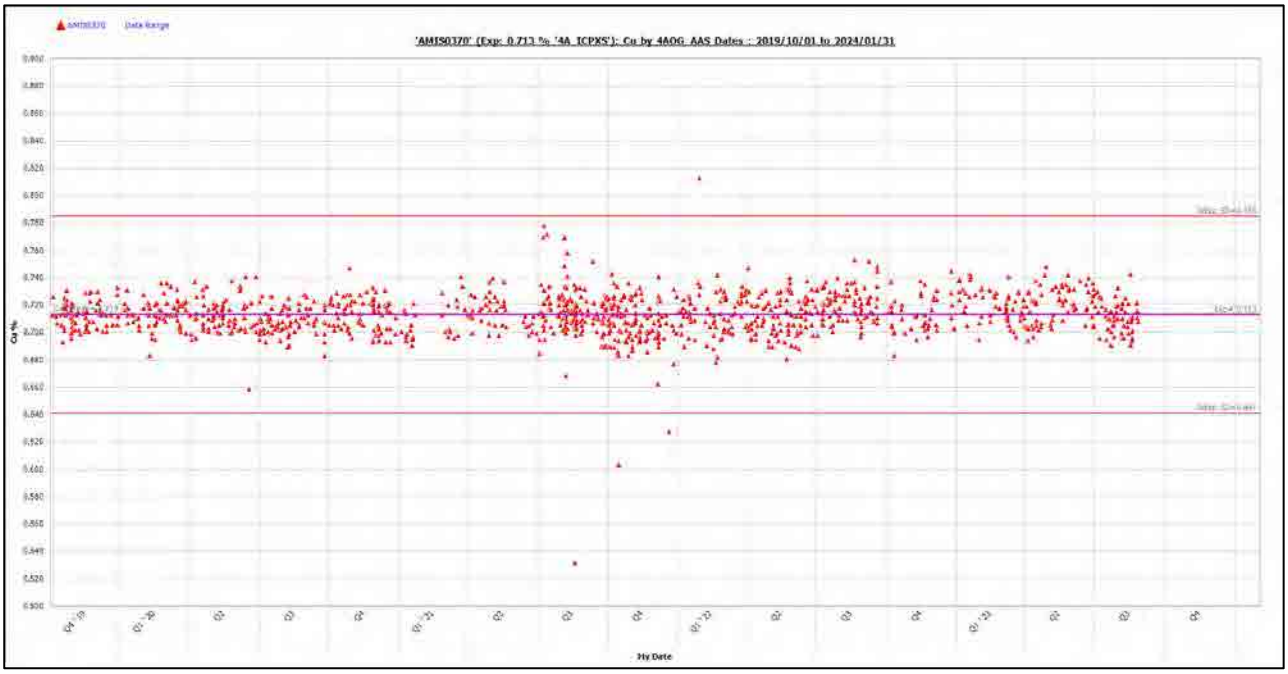


Figure 11-23 RC control chart for AMIS 0474 – TCu by instance - four acid digest with AAS finish (2019-2024)

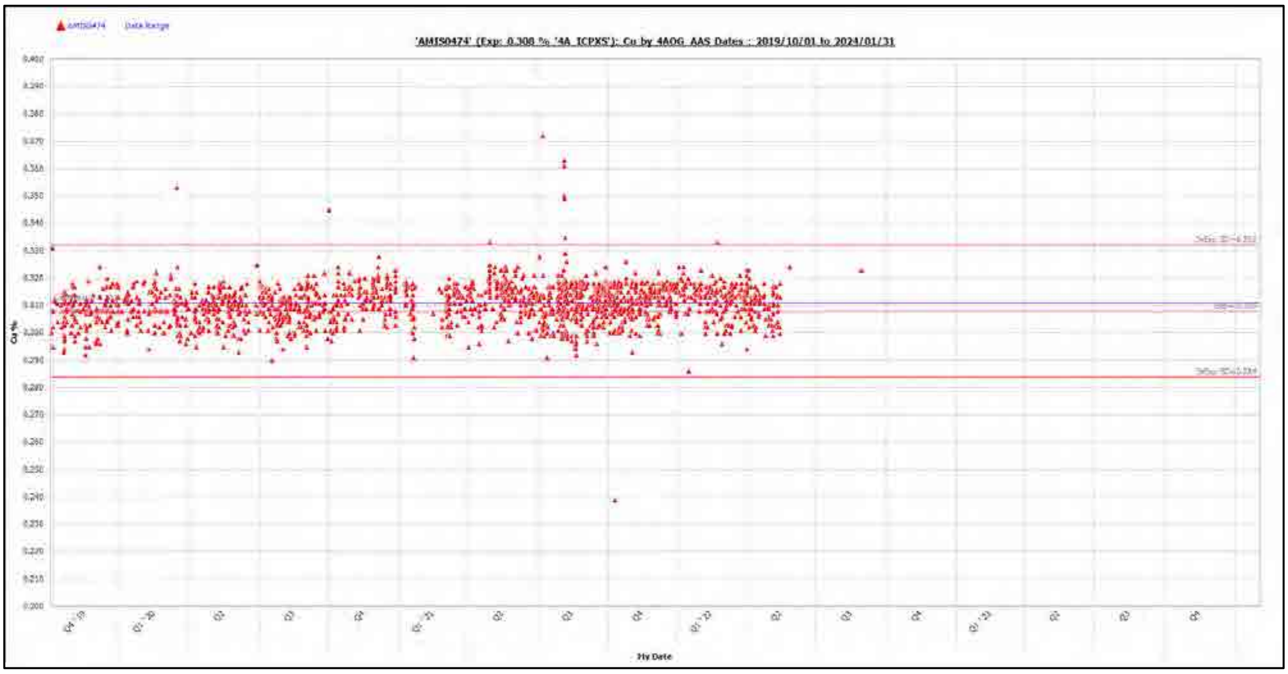


Figure 11-24 RC control chart for AMIS 0795 – TCu by instance - four acid digest with AAS finish (2019-2024)

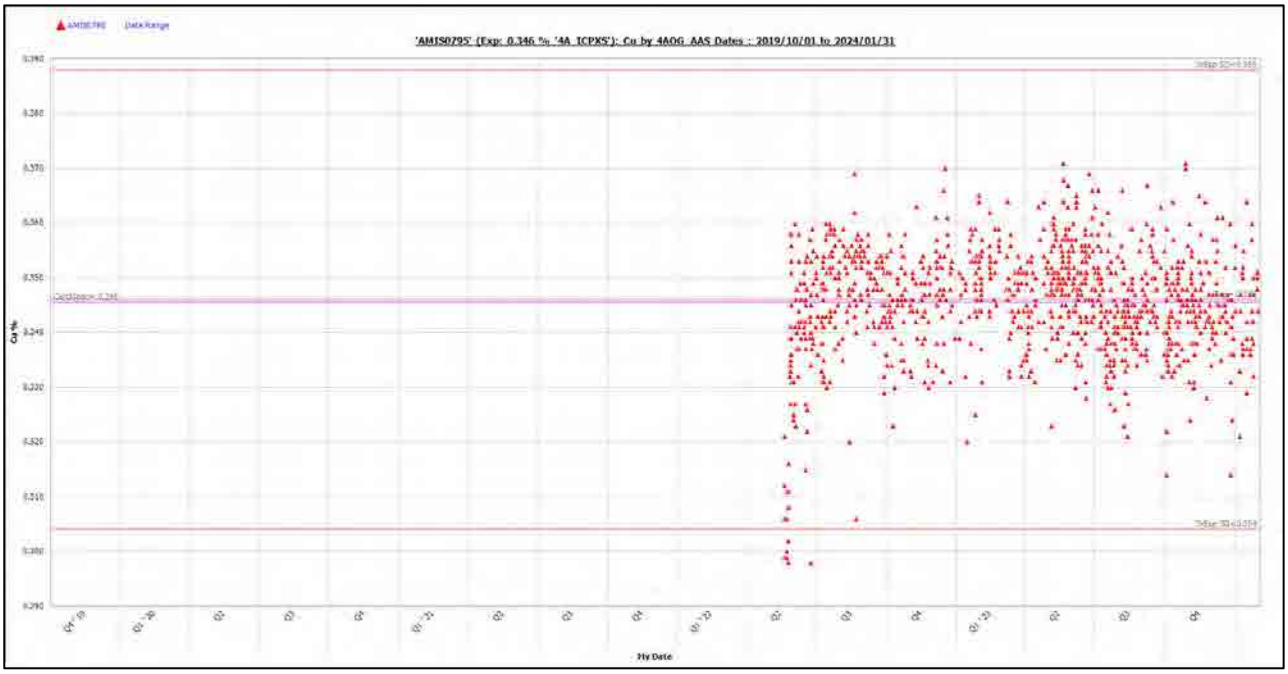
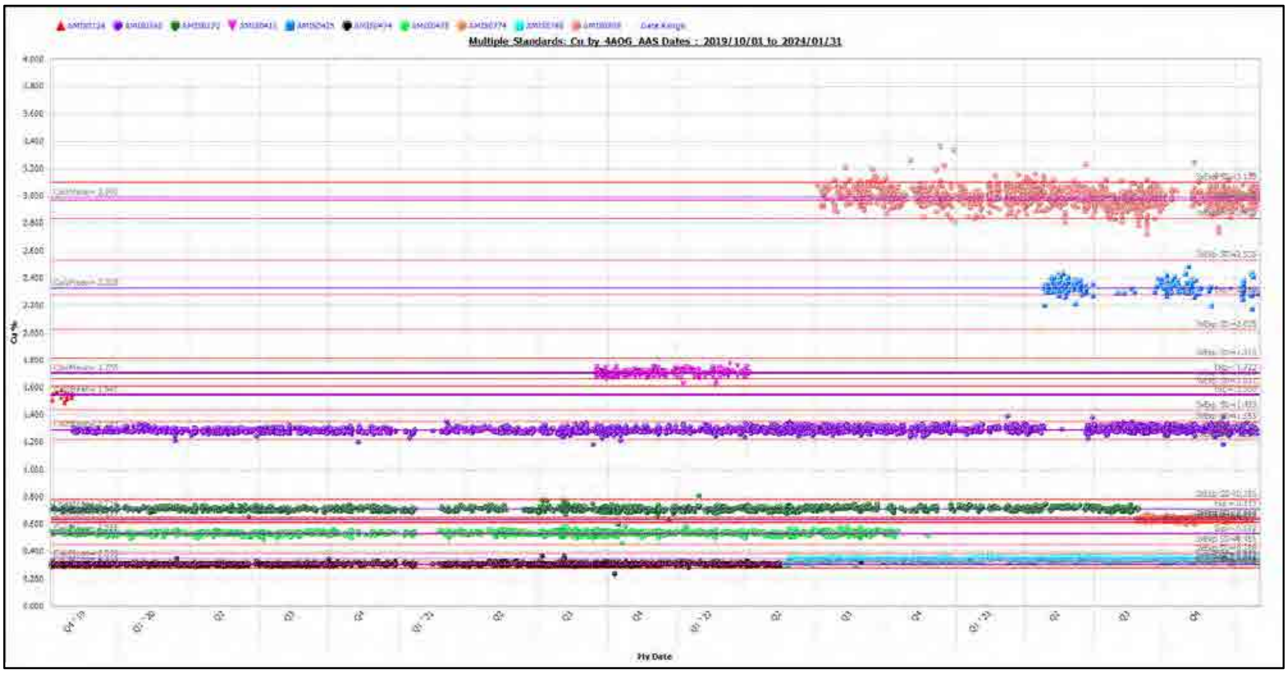


Figure 11-25 RC control chart for multiple CRMs – TCu by date - four acid digest with AAS finish (2019-2024)



Failures resulted from sample swaps, transcription errors and analytical inaccuracies. These failures were investigated and where necessary, batches were re-assayed. The results demonstrate analytical accuracy for both diamond core and RC analysis.

11.9.3 Blank sample analysis and results

Different blank types (coarse and powder) were inserted to monitor contamination and sample swaps. Analysis of the blank data indicated good control over contamination and swaps. The DD blank failure rate between 2019 and 2024 was 1.0%, while for RC samples the failure rates were 0.2% for ALS_KAN and KMP_KAN, indicating improvement. Figure 11-26, Figure 11-27, Figure 11-28 and Figure 11-29 summarise the blank performance between 2019-2024.

Figure 11-26 RC Coarse Blanks control chart by date (2019-2024)

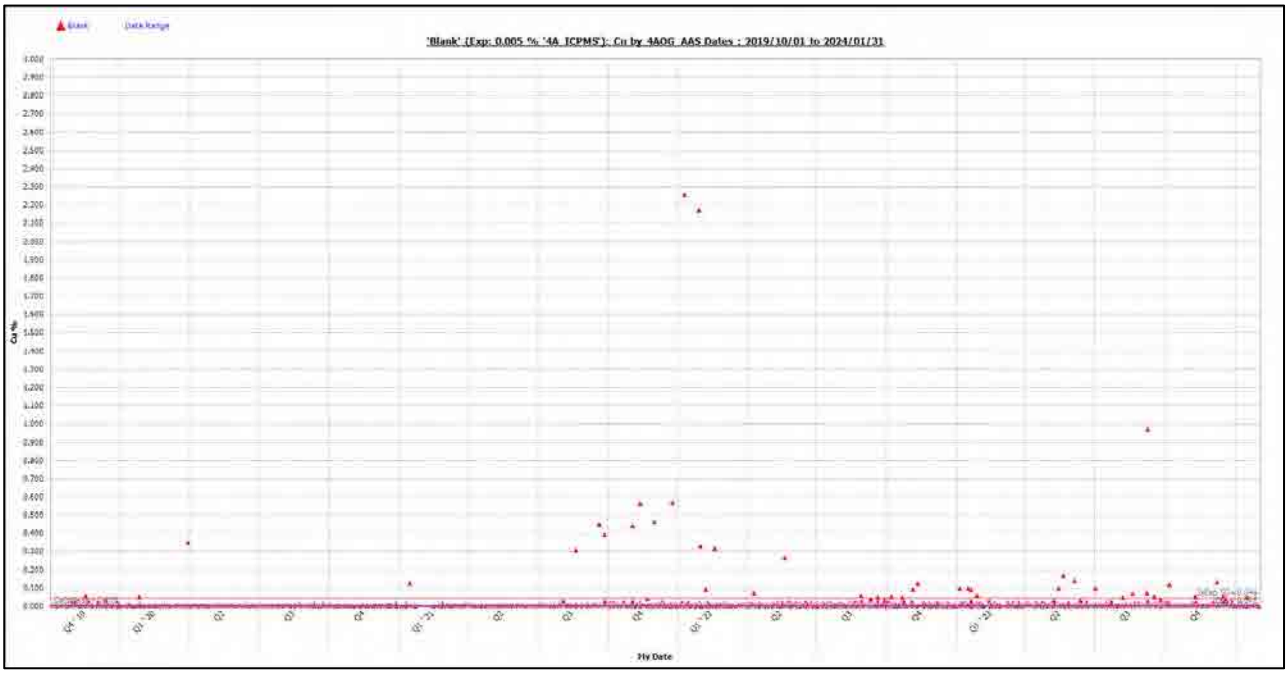


Figure 11-27 DD Coarse Blanks control chart by date (2019-2024)

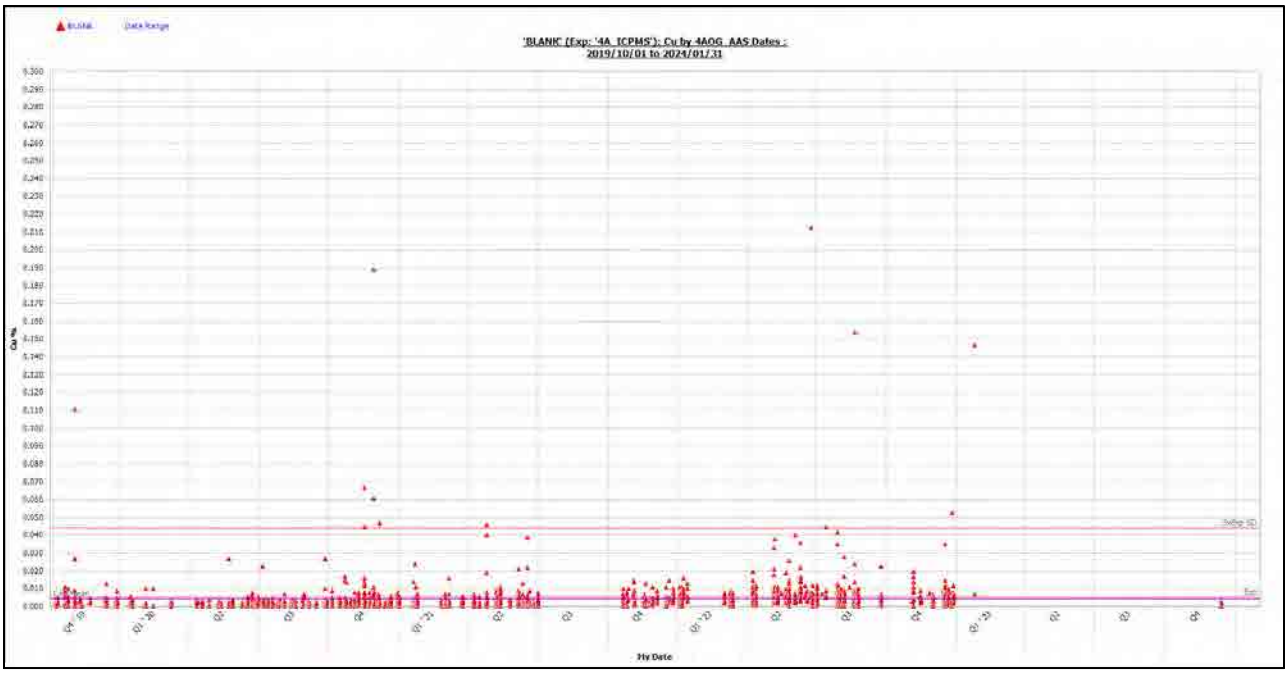


Figure 11-28 RC Pulp Blanks control chart by instance (2019-2024)

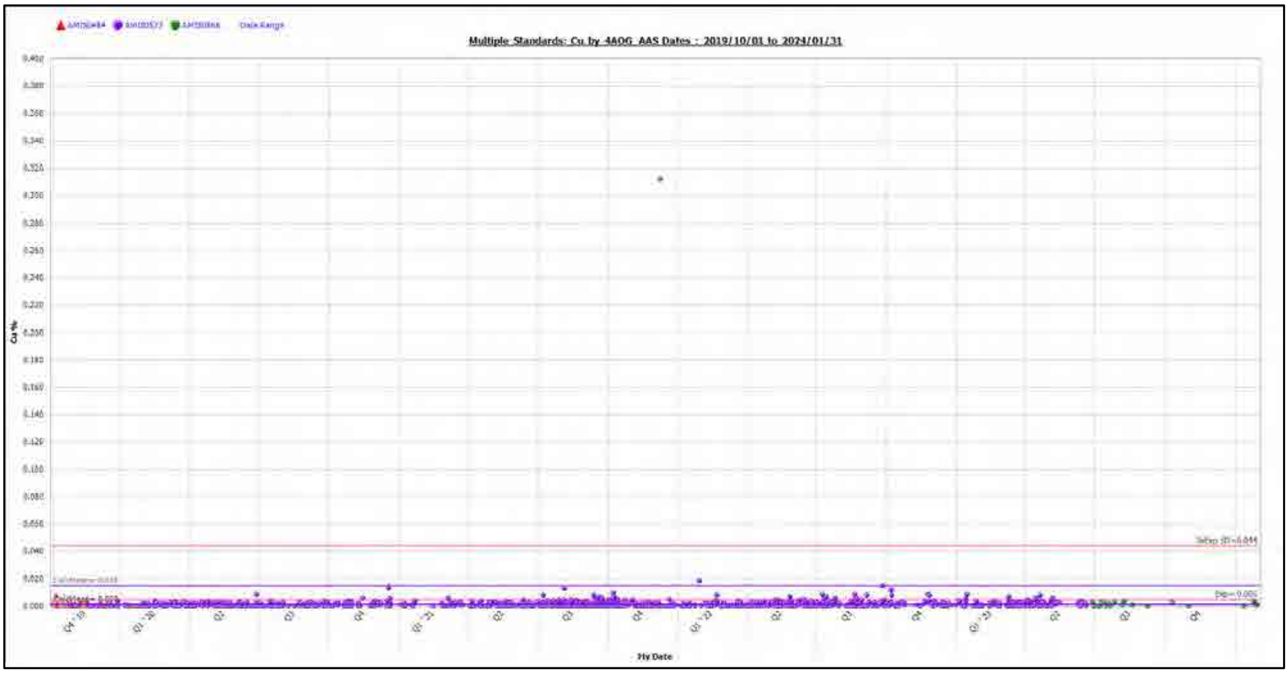
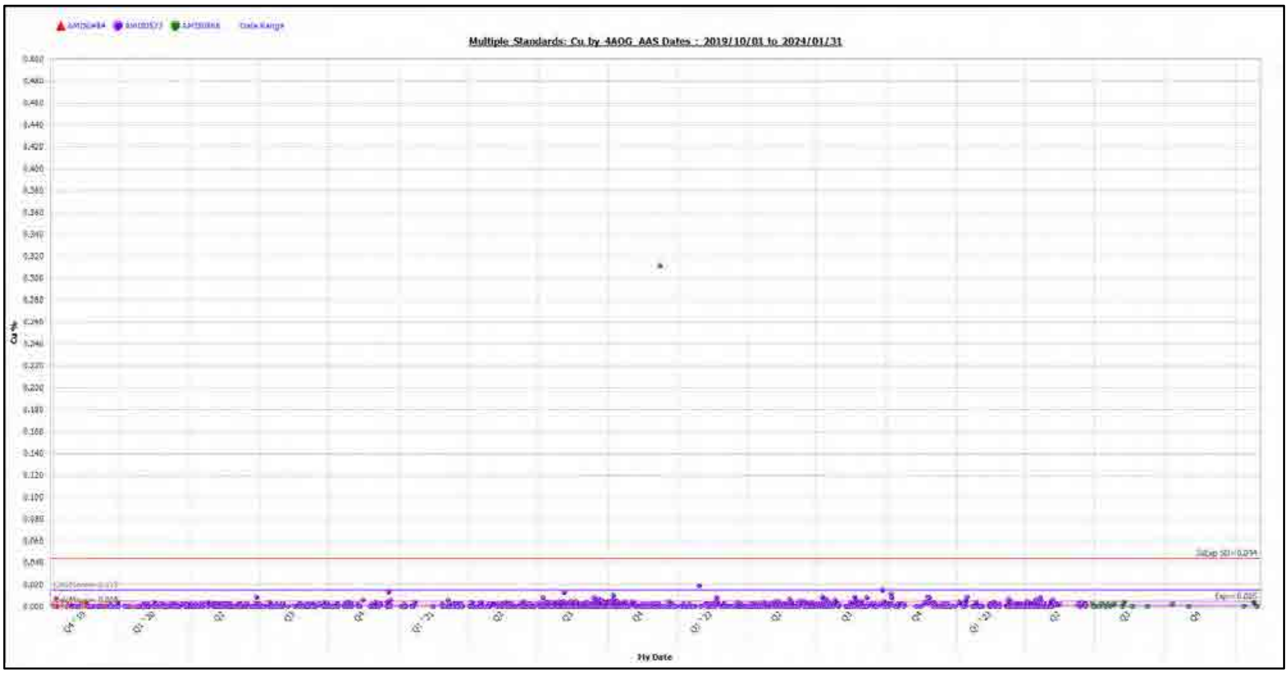


Figure 11-29 DD Pulp blanks control chart by instance (2019-2024)



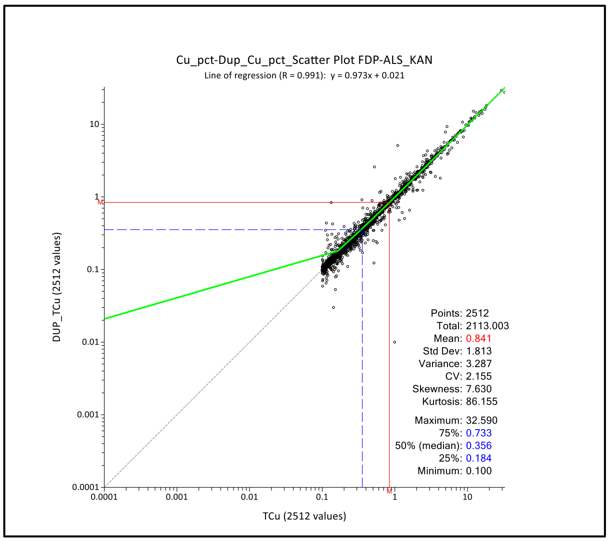
DD blank failures where predominantly due to analytical errors, while RC blank failures were a combination of sample swaps and analytical errors. Failures were investigated, sample swaps corrected and analytical failures were re-assayed as necessary. Failure rates for both DD and RC were well within acceptable limits, and the risk of contamination was low.

11.9.4 Duplicate sample analysis and results

RC field duplicate (FDP) samples were introduced in 2015, with modifications to the RC drill cyclones in 2016 allowing for direct collection of FDPs. This remains routine QAQC practice for all RC drilling, ensuring sampling consistency and precision at the cyclone sample collection stage. Analysis of FDP results indicate good performance with positive correlations for RC chip samples for both ALS_KAN and KMP_KAN (Figure 11-30 and Figure 11-31). The analysis showed that 90% and 96% of ALS_KAN and KMP_KAN sample values fell within ≤ 10% HARD.

In 2023, riffle duplicate (RDP) samples were introduced to test the precision of the riffle splitting stage following sample collection from the RC cyclone. RDPs are collected from a 50:50 split off a 2 tier Jones riffle splitter and now form part of standard QAQC practices, monitored over time. RDP performance to date has shown acceptable results, with 96% of KMP_KAN and 79% of ALS_KAN RDPs within ≤ 10% HARD (Figure 11-32 and Figure 11-33).

Figure 11-30 Scatter and HARD plots for RC field duplicates (FDP) at ALS_KAN (2019-2024)



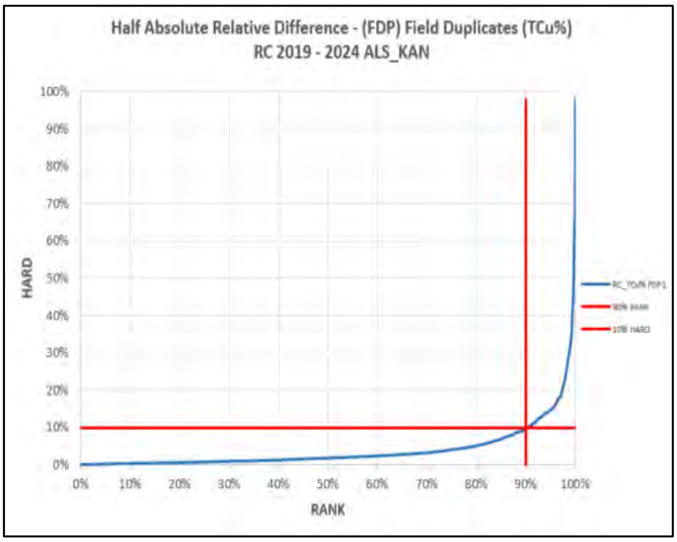
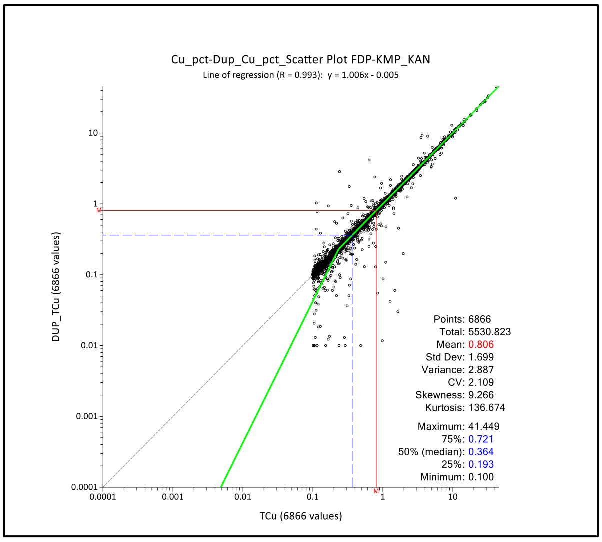


Figure 11-31 Scatter and HARD plots for RC field duplicates (FDP) at KMP_KAN (2019-2024)



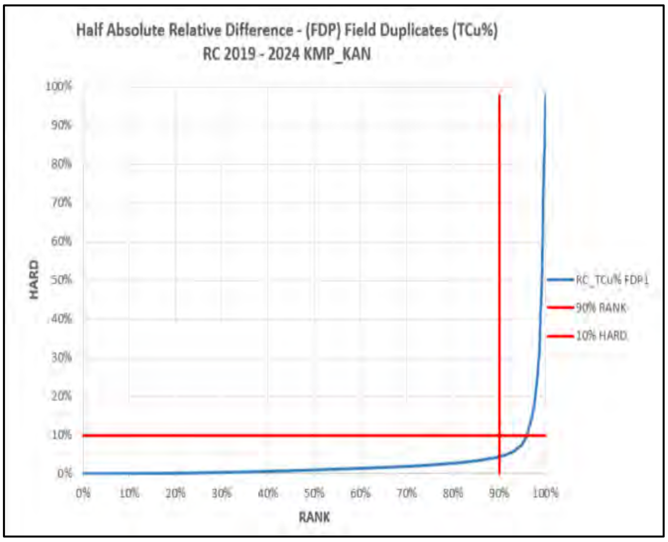
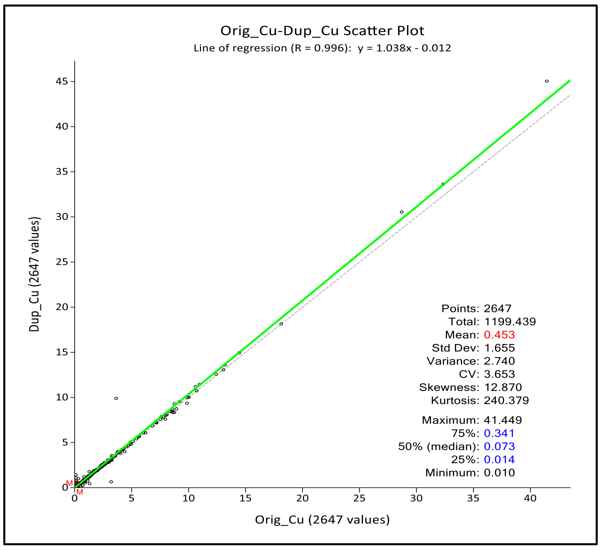


Figure 11-32 Scatter and HARD plots for RC Riffle Split duplicates (RDP) at ALS_KAN (2019-2024)



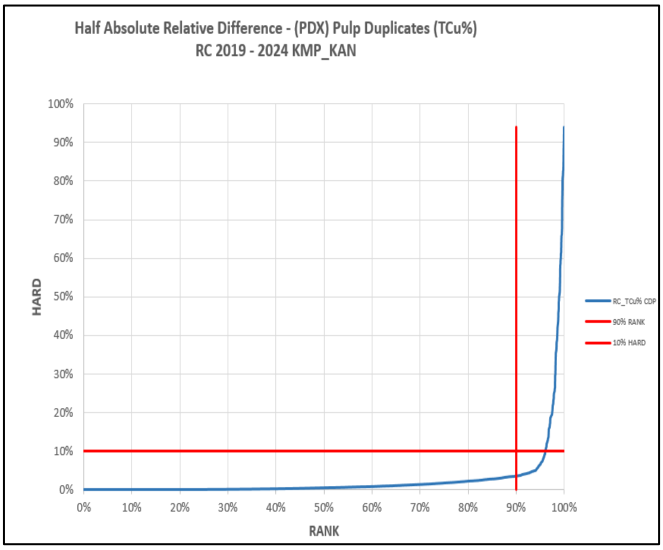
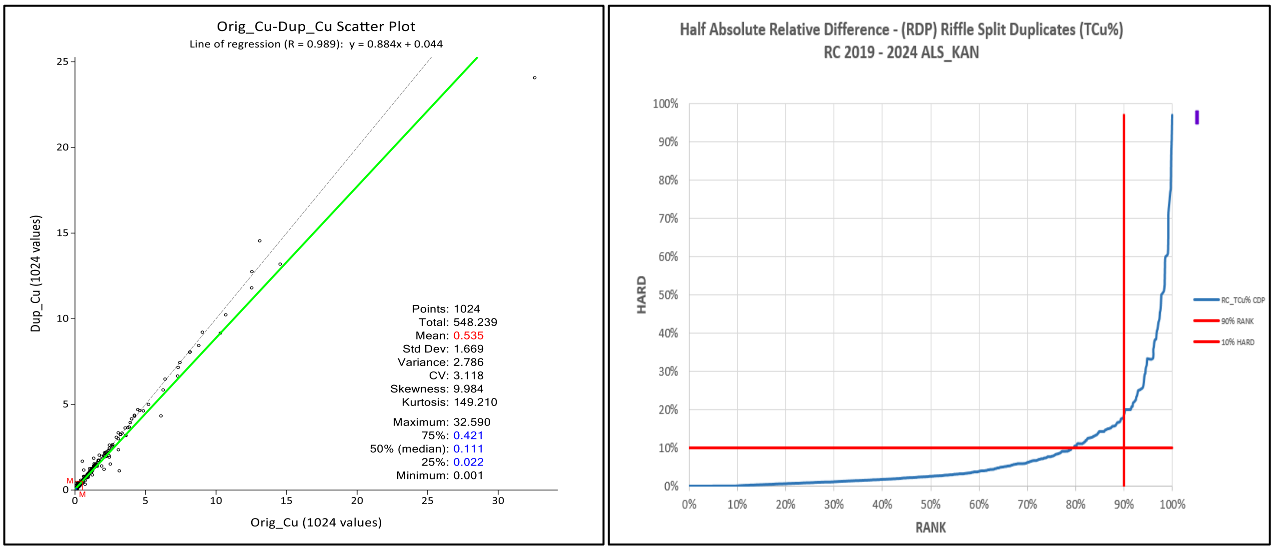


Figure 11-33 Scatter and HARD plots for RC Riffle Split duplicates (RDP) at KMP_KAN (2019-2024)



Coarse duplicate (CDP) samples were taken from the coarse reject fraction after crushing of both diamond core and RC chip samples to test the repeatability of the sample crushing and splitting stage. There was a strong positive correlation between diamond core CDPs and the original samples, performing well, with 95% of the sample pairs within ≤ 10% HARD (Figure 11-34). RC CDPs also demonstrated repeatable sample preparation with 98 % of RC CDPs within ≤10% HARD for both ALS and KMP labs (Figure 11-35 and Figure 11-36).

Figure 11-34 Scatter and HARD plots for DD coarse crushed duplicates (CDP) at ALS_KAN (2019-2024)

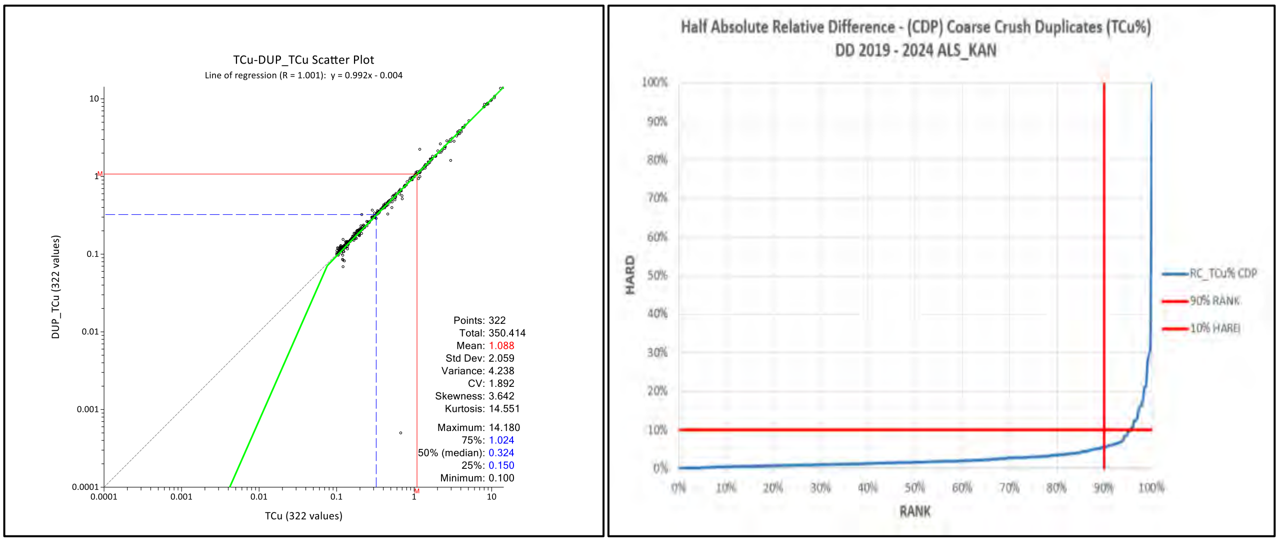


Figure 11-35 Scatter and HARD plots for RC coarse crush duplicates (CDP) at ALS_KAN (2019-2024)

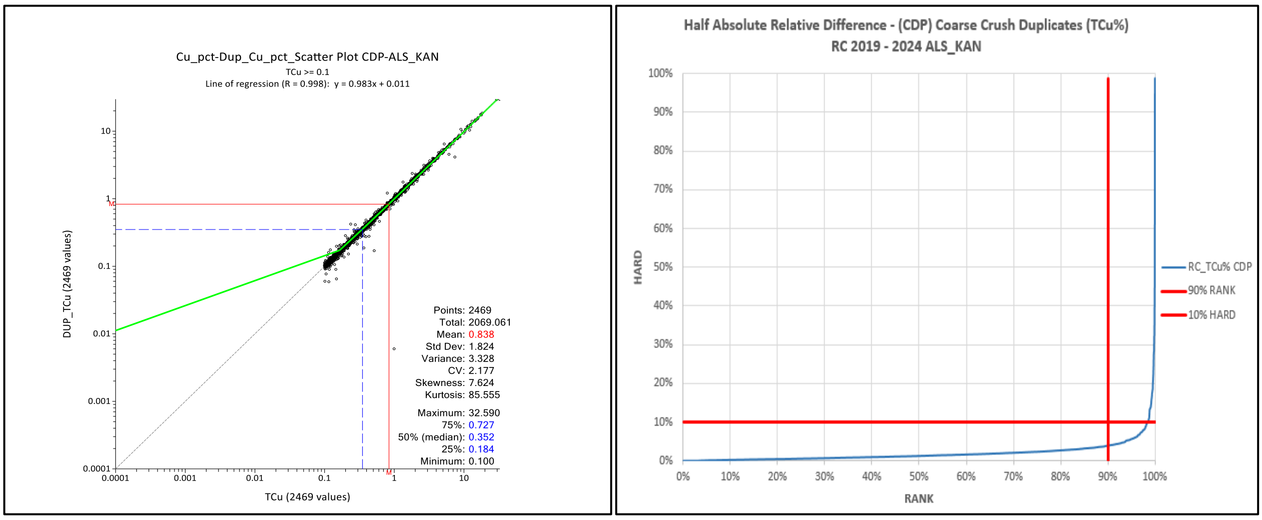
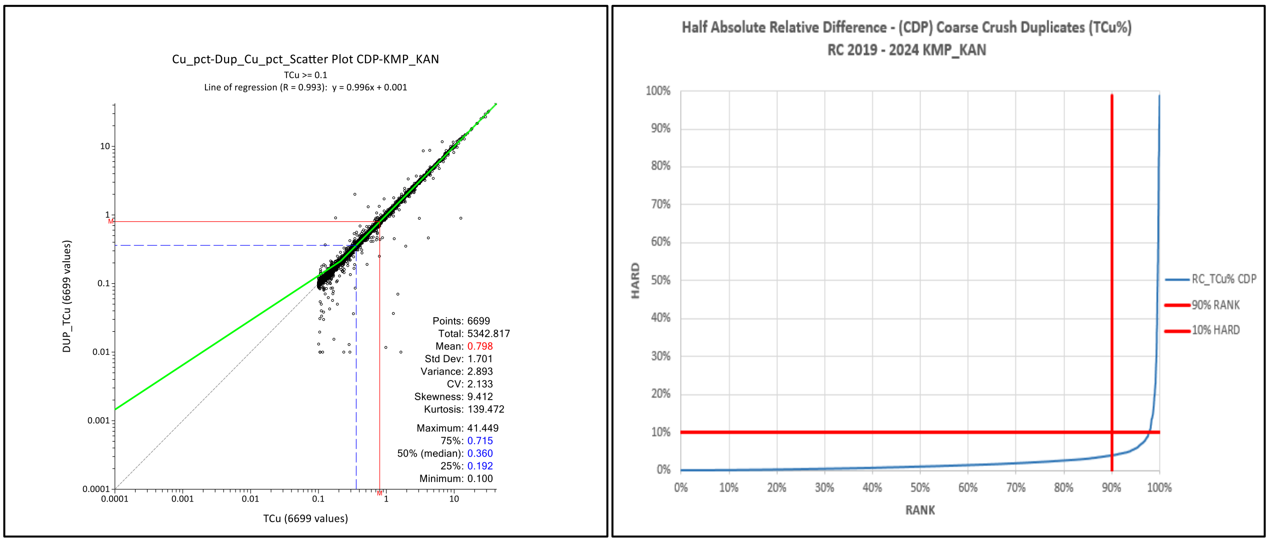


Figure 11-36 Scatter and HARD plots for RC coarse crush duplicates (CDP) at KMP_KAN (2019-2024)



Pulps duplicates (PDP) were inserted to test analytical precision. Scatter and HARD plots for diamond core and RC chip samples indicated that PDPs performed well with good correlation and HARD values within ≤10% (Figure 11-37,

Figure 11-38 and Figure 11-39). For diamond core samples, 98% were within ≤10% HARD, and for RC chip samples, 99% of ALS_KAN data and 98% of KMP_KAN were within ≤10% HARD. Any deviations were investigated and adjustments were made to sampling and analytical practices as needed. No apparent issues with pulverisation or homogenisation were observed.

Duplicate sample coverage was adequate for QAQC assessment of laboratories across sample campaigns and batches.

Results demonstrated that RC chip and diamond core sampling methods are suitable and repeatable for the Kansanshi deposits.

Figure 11-37 Scatter and HARD plots for DD pulp duplicates (PDP) at ALS_KAN (2019-2024)

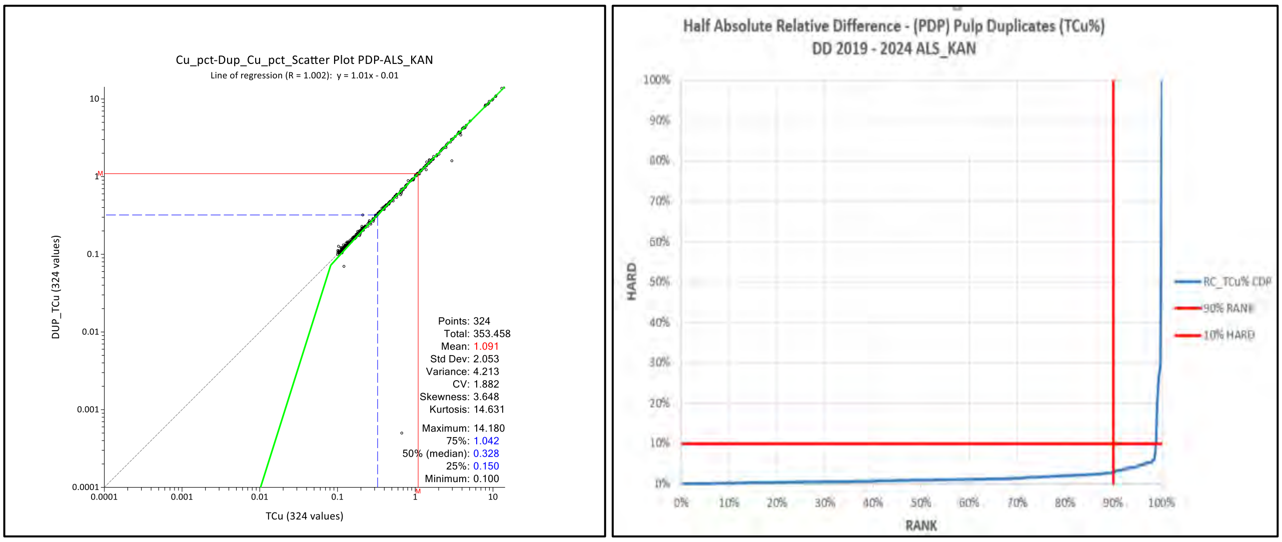


Figure 11-38 Scatter and HARD plots for RC pulp duplicates (PDP) at ALS_KAN (2019-2024)

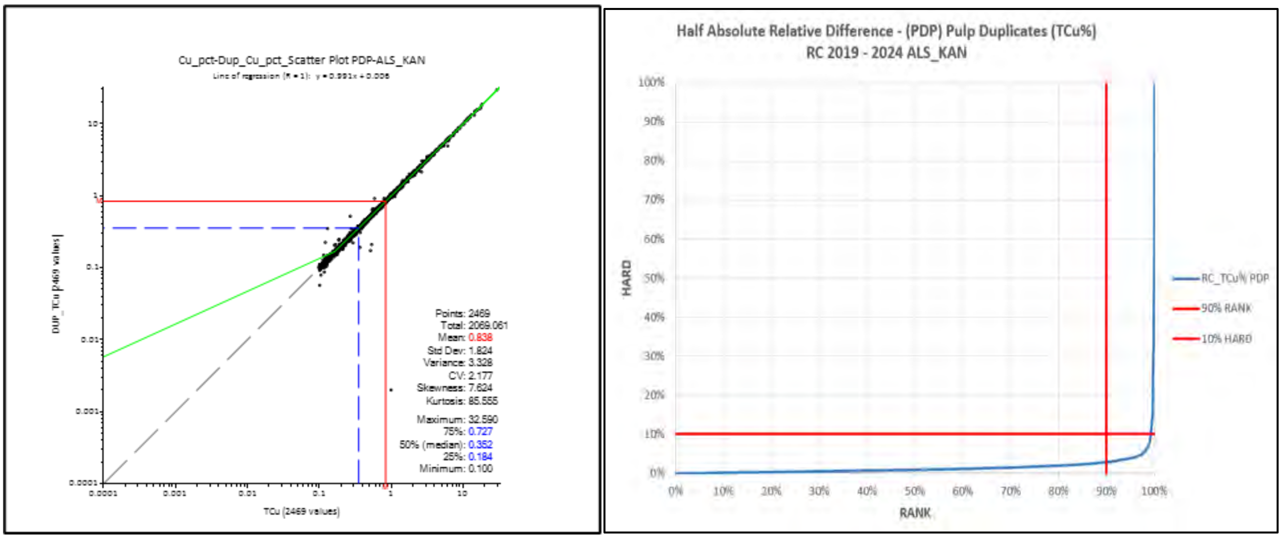
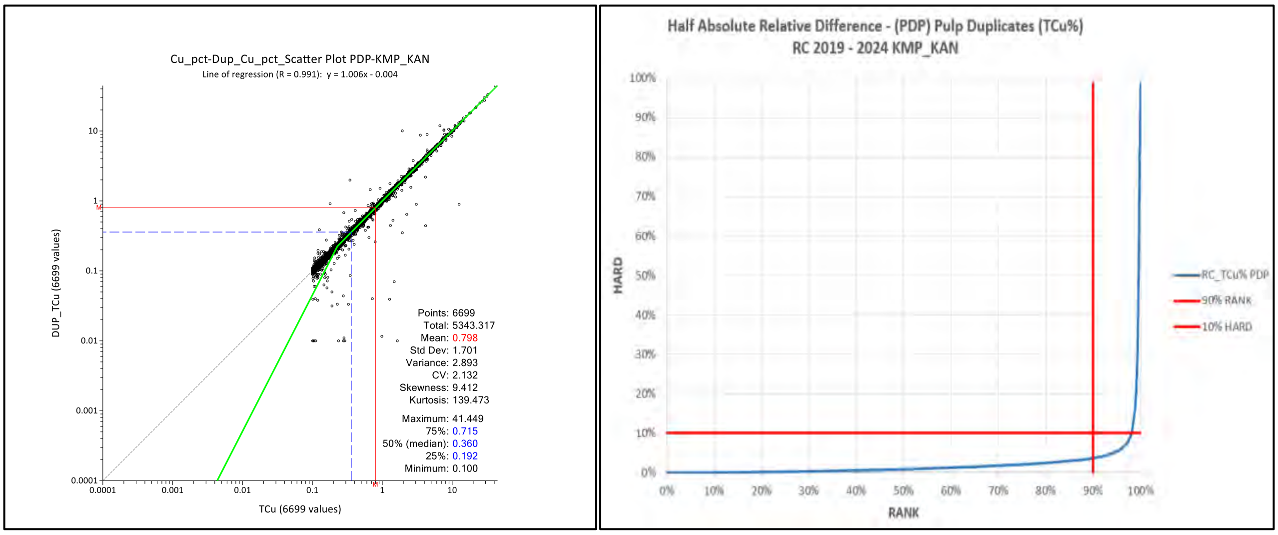


Figure 11-39 Scatter and HARD plots for RC pulp duplicates (PDP) at KMP_KAN (2019-2024)



11.10 Comments on sample preparation, security and analytical procedures

The following key sampling and analysis practices have continued:

- Improved drilling sample quality.
- ALS_KAN's SADCAS accreditation.
- Increased supervision and interaction between analytical and technical team members, along with regular performance review meetings.
- External reviews and ad-hoc visits by ALS technical audit teams to ensure directed management of KMP sampling and analysis practices and results.
- Increased insertion rates for QAQC samples.
- Standardisation and improvement of database systems, analysis and reporting.
- Regular reporting of QAQC results for analysis and performance management.

In the opinion of QP, Carmelo Gomez Dominguez, the sample preparation, security, and analytical procedures for the Kansanshi diamond core and RC chip samples meet industry standard. Sample collection and preparation were conducted using standard equipment to ensure the representativeness of samples of the in-situ geology.

RC chip samples and diamond core are stored securely on-site in locked facilities. Appropriate delivery and sign-off tracking documents have been used for the transportation of samples to the respective laboratories.

Laboratory analytical methods and techniques adhere to industry standards. Combined with QAQC practices, these methods have ensured that sample analytical results are representative, maintaining sample precision, analytical accuracy, and mitigating the risk of contamination. These results are deemed adequate to support the Mineral Resource estimate update of the Kansanshi deposits.

ITEM 12 DATA VERIFICATION

The QP, Mr Carmelo Gomez, has conducted regular site visits to the Kansanshi mine since 2018, with the most recent visit in June 2022. Prior to this, Mr Gomez worked part-time on-site. During these recent visits, Mr Gomez inspected the respective aspects of data collection associated with diamond drill core and RC chip sampling, along with relevant QAQC results. He reviewed the quality of in-pit mapping data and confirmed that the methods used for compiling geological strata and vein data, and their modelling, honoured the input data, ensuring the models are representative of the prevailing geology. Detailed reviews and analysis of production reconciliation were also completed to support the accuracy of resource and ore control model estimates as per the assigned Mineral Resource classification.

Contributing authors, Mr David Gray (Group Manager, Mine Geology and Resources, FQMA, Australia) and Mr Louis Van Heerden (Group Principal Geologist, Mines, FQM, South Africa), along with KMP mine geologists, supported the QP in these data verifications. Verifications included assessing the integrity of diamond core sampling, bulk density estimation, sample preparation and dispatch procedures, QAQC practices, in-pit mapping and 3D geology modelling. Specifically, verifications have included:

- Site visits verified quality drilling, sampling, preparation, analytical and QAQC practices.
- Verification of RC chip sampling QAQC confirmed adherence to industry standards. Results demonstrated that sample results are repeatable, accurate and have effectively contained contamination.
- Diamond core sampling QAQC practices were verified as comprehensive and in line with industry standards. QAQC results indicated precise and accurate results with effective contamination control.
- Diamond core and RC chip samples are stored in fenced, safe and secure covered facilities with restricted gate access on the mine site.
- Investigation into bias risks between samples using different drilling methods showed marginal to no definitive risk of bias between RC and diamond drilled samples.
- In-pit observations of prevailing geology confirmed accurate representation in the current geological model.
- Verification of diamond and RC drill collar coordinate measurements and visual checks against digital database data confirmed accuracy.
- Documented sampling and sample preparation methods were reviewed and found to be correct when compared with those stored in the database.
- Verification of a small percentage of actual laboratory assay records against database data revealed no transcription errors.
- The majority of diamond core analysis was conducted at ALS_KAN and ALS Johannesburg with good insertion rates for umpire checks (Genalysis Johannesburg). Umpire results verified the accuracy and precision of primary laboratory analyses.
- Diamond drilling data and the RC grade control data are hosted in SQL databases, ensuring that data is validated as it is captured. The SQL database data was reviewed by the QP, with no significant issues observed.

In the opinion of the QP, Mr Carmelo Gomez, the KMP data is of good quality for use in Mineral Resource estimation. The data has sufficient coverage of the respective domains of mineralisation and has been collected, sampled and analysed with secure storage practices. Sample assay data is believed to be of good quality due to robust sampling methods and effective QAQC practices. Continued RC ore control QAQC and management of bias risks, along with enhanced data validation, will further increase confidence in future data collection and resulting estimates.

ITEM 13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Mineralogy

Mineralisation at Kansanshi is associated with veins, select lithologies and occasional fault breccia zones. Styles of mineralisation are mainly oxide, non-primary sulphide and primary sulphide, and with distinct zones of transitional/mixed oxide and sulphide mineralisation.

Primary sulphide mineralisation is dominated by chalcopyrite with minor bornite and some gangue pyrite, the latter being more prevalent in plant feed from the North West Pit. The dominant oxide mineral assemblages are malachite and chrysocolla. Secondary sulphides are predominantly chalcocite.

Limited refractory mineralisation, which typically outcrops at surface and is common in the saprolite horizon, is not recoverable in the Kansanshi processing circuits.

13.2 Mineral processing overview

The mineral processing facilities at Kansanshi have undergone several periods of expansion since the achievement of commercial production in 2005. Item 6 provides a brief historical summary of these expansions, whilst the 2024 status of the mineral processing facilities is described more fully in Item 17.

The current capacity of the processing facilities is approximately 7 Mtpa of oxide ore, 8 Mtpa of mixed ore and 13 Mtpa of sulphide ore. Further proposed expansions to the sulphide processing capacity (an additional 27.5 Mtpa) and the on-site smelting capacity (from nameplate 1.2 Mtpa, up to 1.6 Mtpa) for Kansanshi and Sentinel copper concentrates are described in Item 17 of this Technical Report.

13.2.1 Expansion projects

Leaching and smelting expansion synergy

A 2013 leach capacity expansion to 12 Mtpa included the construction of additional flotation, leaching, counter current decantation (CCD) thickeners, solvent extraction (SX) and electrowinning (EW) facilities to increase the copper cathode production rate. Milling of the additional plant feed 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 an on-site smelter at Kansanshi. The rationale behind this construction arose from the shortage of smelting capacity in Zambia at the time, exacerbated by the imminent commencement of concentrate production from the Company’s Trident Plant, processing mined ore from the Sentinel Pit.

The first concentrate was smelted at Kansanshi in March 2015. The smelter also produces significant quantities of sulphuric acid as a smelting by-product. This acid can be consumed by the leaching of the remaining oxidised ore from the mine. Furthermore, the ready availability and reduced cost of the smelter acid allows a significant proportion of mined ore classified as Mixed to be economically treated by leaching.

S3 Sulphide Expansion Project

The S3 Sulphide Expansion Project is an expansion to the existing Kansanshi processing facilities. The expansion includes the construction of a new copper concentrator capable of treating an additional 27.5 Mtpa of sulphide ore. The design for the S3 expansion facilities has been based on that previously designed for the Company’s Sentinel processing plant, although taking into account metallurgical and site-specific conditions that differ from the Sentinel plant feed.

An updated design throughput of 27.5 Mtpa (the capacity of one of the two processing trains at Sentinel) has been adopted for the S3 expansion, but with the following fundamental differences from the Sentinel flowsheet noted:

- Kansanshi sulphide ore has a free gold content requiring gravity recovery
- Kansanshi sulphide ore does not require flash flotation
- the ore contains high levels of organic carbon and pyrite which will require depression to produce acceptable concentrate grades
- a shorter rougher flotation residence time will suffice, followed by multiple cleaning stages including conventional cells and flotation columns
- rougher concentrates will be pumped to the main plant for cleaning in a new circuit, along with the S2 concentrates
- reagents required in the S3 plant will comprise collector, frother and NaHS for flotation, CMC for carbon depression, and lime for pyrite suppression

In the longer-term production plan for Kansanshi, sulphide ore from the South East Dome Pit will become the predominant feed type, thereby necessitating the sulphide processing capacity expansion in order to maintain copper metal production output levels.

13.3 Metallurgical testwork

13.3.1 Existing plant feed types

Predicted metallurgical performance for Kansanshi ores relates to a relatively long period of operational experience, rather than to testwork that was carried out in immediate years past. An explanation of the manner in which processing recovery is determined and projected is provided in Item 13.4.

13.3.2 S3 plant feed types

In 2017, about 100 drill core samples from South East Dome were submitted to Petrolab in the United Kingdom for detailed mineralogical examination. Upon completion of this examination, selected samples were subjected to further metallurgical testwork. Relying on the limited testwork results, the similarities to the ore from the Main Pit, experiences gained from processing sulphide ore via the existing S2 circuit, and more stringent concentrate quality requirements, several modifications were required to the flowsheet proposed for S3. In addition, the S2 cleaner circuit modifications were brought forward to test the modified circuit and reagents, which continue to be optimised. The testing of this circuit has been used to inform the final S3 cleaner flowsheet.

These modifications include the installation of flotation columns and Jameson cells for concentrate cleaning, inclusion of controlled potential sulphidisation (CPS) for partially oxidised ores and secondary sulphides, inclusion of a lime dosage system for the depression of pyrite and the potential addition of other reagents for the depression of carbon. In terms of comminution, copper sulphide minerals from South East Dome are coarsely grained, and hence the grind size has been relaxed to a P_{80} of more than 180 – 212 µm for S3.

In 2023, additional South East Dome samples were sent to Process Mineralogical Consulting in Canada. These were sampled by lithology and submitted for characterisation of hardness as well as naturally flotable gangue at base pH and elevated pH. The hardness testwork indicated a potential risk associated with some key lithologies (i.e., carbonaceous knotted schist and carbonaceous phyllite) being harder than expected. To understand this further, 82 additional samples from South East Dome and new Main pit cut-backs are now (in Q1 2024) being subjected to SAG Mill Comminution, Bond Ball mill, and Hardness Index Testing for Axb and Bond work index measurements. In this case, the hardness characteristics will be linked to the lithology and alteration characteristics of the ore to inform applicability/ extrapolation of the results throughout the volume of the deposit.

As noted above, 100 drill core samples from South East Dome were submitted for detailed mineralogical study in 2017, which indicated similarities to the plant feed currently being treated from the Main Pit. Several samples were submitted for additional metallurgical testwork, which informed some of the decisions regarding modifications to the S3 circuit design. Samples taken for metallurgical testwork in 2023 were provided based on the various lithologies present in the deposit. Carbonaceous phyllite samples from the South East Dome were identified as being similar to material from the Main Pit currently being treated. Additional testwork on these samples and on other distinct Kansanshi ore types is currently being undertaken at BLM in Canada.

It is concluded that to the best of the QP's understanding, all ore types and mineralisation styles in the South East Dome deposit have been sampled and subjected to the appropriate testwork to define the S3 circuit design and optimum operating conditions.

The 2023 flotation testwork reaffirmed the need for a solution to reject both organic carbon and pyrite in S3; organic carbon, iron and sulphur recoveries were high for the dominant copper bearing lithologies in South East Dome. Copper recoveries were good, even at low grades. A sample of carbonaceous phyllite ore from the Main Pit that appears typical of carbonaceous phyllite in South East Dome, along with other distinct Kansanshi ore types, is undergoing flotation optimisation testwork at Base Metallurgical Laboratories during Q1 2024.

Other solutions to reject activated pyrite and carbon are being investigated through plant trials on S2, and laboratory testwork at KMP, and at Grinding Solutions in the United Kingdom. These include specialised pyrite depressants (Kansanshi, plant and laboratory) and gravity separation methods to reject carbon from final concentrate (Grinding Solutions).

13.3.3 Other testwork

Other than the testwork on South East Dome ores, no major testwork programmes have been undertaken in the last few years in relation to the existing processing facilities. Some confirmatory testwork has been undertaken, however, to support and optimise existing operations, i.e.:

- optimisation of leach temperature in the oxide leach circuit
- testwork and plant optimisation leading to the addition of CPS to the sulphide flotation circuit
- optimisation of the oxide and mixed ore classifications
- gravity gold recovery work, evaluating locations for additional centrifugal concentrators
- improvement in concentrate grades to maximise smelter capacity, and leading to the inclusion of a Jameson cell and columns in the cleaner circuit.

13.4 Processing recovery

The ore processing route at Kansanshi is determined by the relative proportions of acid soluble copper (ASCu) and acid insoluble copper (AICu) in the mined ores, where total copper (TCu) equals AsCu + AICu. The ore is classified into three ore types according to 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 in the flotation circuit.
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 non-primary sulphide minerals with minor acid leachable minerals. This material is referred to as Mixed in the Mineral Resource model.

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 be treated economically and is stockpiled.

The sulphide ore is treated by conventional flotation enhanced with CPS (i.e. conditioning with sodium hydrosulphide, NaHS) mainly to treat tarnished sulphide minerals more effectively. The mixed and the oxide ore types are both treated by flotation to recover acid insoluble and relatively small amounts of oxide copper minerals. Both mixed and oxide flotation circuits are equipped with CPS which helps with the recovery of tarnished and secondary copper sulphides.

All of the tails from the oxide float circuit are directed to leaching. Leaching is followed by SX and EW to produce copper cathode. A portion of the tails from the mixed float are also directed to acid leaching whilst the remainder is directed to final tails. The decision to leach mixed float tails is an economical decision driven by the sale price of acid balanced against the gangue acid consumption of the mixed float tails, and the additional copper recovery achievable.

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 to be economically treatable by leaching. As such, the transition between mixed float and mixed leach ore remains flexible, determined by economics and ore availability.

The three different ore types at Kansanshi have differing metallurgical recovery responses. These responses vary according to grade attributes and hence there is no one single recovery value that is applicable to each ore type. Variable processing recovery projections are therefore based on KMP's analysis of actual production, and detailed analysis over time has shown that the recoveries for each of the process routes are non-linear. The form of these variable recovery equations (or expressions) can be summarised as follows:

1. Total plant recovery: The applicable equation is based on accurate measurements of copper in concentrate, cathode and feed, and uses inputs from weightometers, samplers and chemical assays. This particular equation has been used for month-end recovery calculations since January 2016 and is believed to be an accurate representation of copper recovery at Kansanshi. This form of equation is also used for month-end and external reporting.

$$Total\ plant\ recovery = \frac{Copper\ in\ concentrate\ and\ cathode}{Copper\ in\ feed} x100$$

2. Individual flotation circuit recovery: In this equation, recovery is calculated via a two-product formula that is based on feed, concentrate and tails assays. It is used for the calculation of individual flotation circuit recoveries on a daily basis for process control and internal production reporting purposes.

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

3. Predicted recovery models: Statistical recovery models are updated for each circuit on an annual basis to reflect ore feed changes and process improvements. These equations are based on TCu and AsCu assays of the plant feed. They are used for automatic recovery target setting, based on TCu and AsCu assays at a point in time. They are also used for production planning and budget setting processes. The recovery equations are adjusted on an annual basis, using actual plant operating results. Normally, about 30 months of data will be used in the analysis, but if a change in operating parameters or reagent

type has resulted in improved recoveries over a three-month period, the equations will also be adjusted accordingly.

$$Predicted\ Flotation\ Recovery = \left(a\left(\frac{ASCu}{TCu}\right) + b\ \left(\frac{AICu}{TCu}\right) + c(TCu - TCu_{Mean})\right)$$

where a, b, and c are constants, that differ between the three flotation circuits.

The recovery equations have been reviewed and updated from those presented in the 2020 Technical Report. The new equations are presented in Table 13-1.

Table 13-1 Processing recovery equations for 2024

Sulphide Ore Flotation	%Cu rec = 28*ASCu/TCu + 92.5*AICu/TCu + 2.1*(TCu - 0.82)
Mixed Ore Flotation	%Cu rec = 20*ASCu/TCu + 88*AICu/TCu + 8*(TCu - 0.94)
Mixed Ore Float-Leach	%Cu rec = 20*ASCu/TCu + 88*AICu/TCu + 8*(TCu – 0.94) + 0.15*% MFT to leach
Oxide Ore Float	%Cu rec = 11*ASCu/TCu + 54*AICu/TCu + 1.95*(TCu - 0.82)
Oxide Ore Float-Leach	%Cu rec = 86*ASCu/TCu + 65*AICu/TCu + 7.21*(TCu - 0.95)

Concentrate grades equations have also been derived from historic operating data:

Sulphide Con Grade (%TCu) = 10.7 * TCu + 16 up to the end of 2024, followed by 24% in 2025, and 25% thereafter

Mixed Con Grade (%TCu) = 12.6 * TCu + 10.2

Oxide Con Grade (%TCu) = 14.3 * TCu + 9

13.4.1 Updated recovery equations for copper and fixed recovery values for gold

Actual recovery performance figures over the 2023 calendar year were:

- an overall average sulphide ore recovery of 87.6%, for an average feed grade of 0.51%TCu
- an overall average mixed ore recovery of 70.7%, for an average feed grade of 0.63%TCu
- an overall average oxide ore recovery of 75.6%, for an average feed grade of 0.83%TCu

Reflective of actual performance during 2023, updated equations were available for 2024, as listed in Table 13-1. After application in the production scheduling process for this Technical Report, the following overall average copper recoveries are apparent:

- primary sulphide ore recovery of 87.7%, for an average feed grade of 0.49%TCu
- mixed ore recovery of 67.7%, for an average feed grade of 0.65%TCu
- oxide ore recovery of 72.8%, for an average feed grade of 0.61%TCu, comprising:
  - a leach recovery of 40.2% for oxide leach ore, and
  - a flotation recovery of 32.7% for oxide flotation ore

Gold recovery into flotation concentrates is expected to be 37.5% for sulphides, 32% for the mixed ore circuit, and 24% for oxides. Additional gold will be recovered through gravity concentrates in each circuit. Overall gold recovery for 2023 was 55% from a head grade of 0.14 g/tAu.

As the mine transitions from oxide and mixed into sulphide horizons, the distribution of gold mineralisation has been reducing. Evidence of gravity gold increasingly reporting to tailings, is a situation which will be remedied with more gravity concentrators being installed and existing circuits being transferred to the sulphide S2 and to the S3 circuits. Hence, it is expected that gold recovery will improve as the sulphide processing facilities are expanded.

The updated copper recovery equations in Table 13-1 are recommended for mine planning purposes.

13.5 Acid consumption

Total acid consumption at Kansanshi is mainly a function of the gangue acid consumption and to a lesser extent acid soluble copper grade. The economic evaluation for the Kansanshi 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 oxide ore.

This is not the case for Mixed Ore where acid price is used to determine whether mixed float tails are leached or not. Gangue acid consumption, copper recovery, and the value of acid sales all contribute to the economic decision of whether it is best to treat the mixed float tails to recover the remaining copper or sell the acid.

ITEM 14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The Kansanshi Mineral Resource estimate (MRE) was completed in February 2024 by Mr Carmelo Gomez, who meets the requirements of a Qualified Person (QP). This estimate covers the three vein hosted and stratiform copper deposits within the Kansanshi mining area, namely the North West, Main and South East deposits.

A 3D block model of key geology domains forms the basis of estimates. Mineralised volume and grade estimates were calculated using geostatistical applications from commercial mining software, including Datamine Studio RM v.2.0.66.0, Snowden Supervisor v.8.14 and Leapfrog Geo v2023.2.1. The effective date of this Mineral Resource estimate is 31st of December 2023.

14.2 Coordinate system

The Kansanshi Operation's license is located in the ARC 1950 (Zambia) UTM 35 South coordinate system. However, coordinates have been transformed into a local mine grid system (Table 10-8). As a results, the coordinates for drill hole data, topographic and pit survey data (digital terrain models, or DTMs) and the Mineral Resource estimate block model, are all recorded using the local mine grid system.

14.3 Data

Since the 2019 MRE included in the 2020 Technical Report, a total of 215 diamond holes, amounting to 69,619 m have been added to the Main, North West and South East deposits. The total number of diamond holes providing data used in this MRE update is 2,220 holes for 740,709 m.

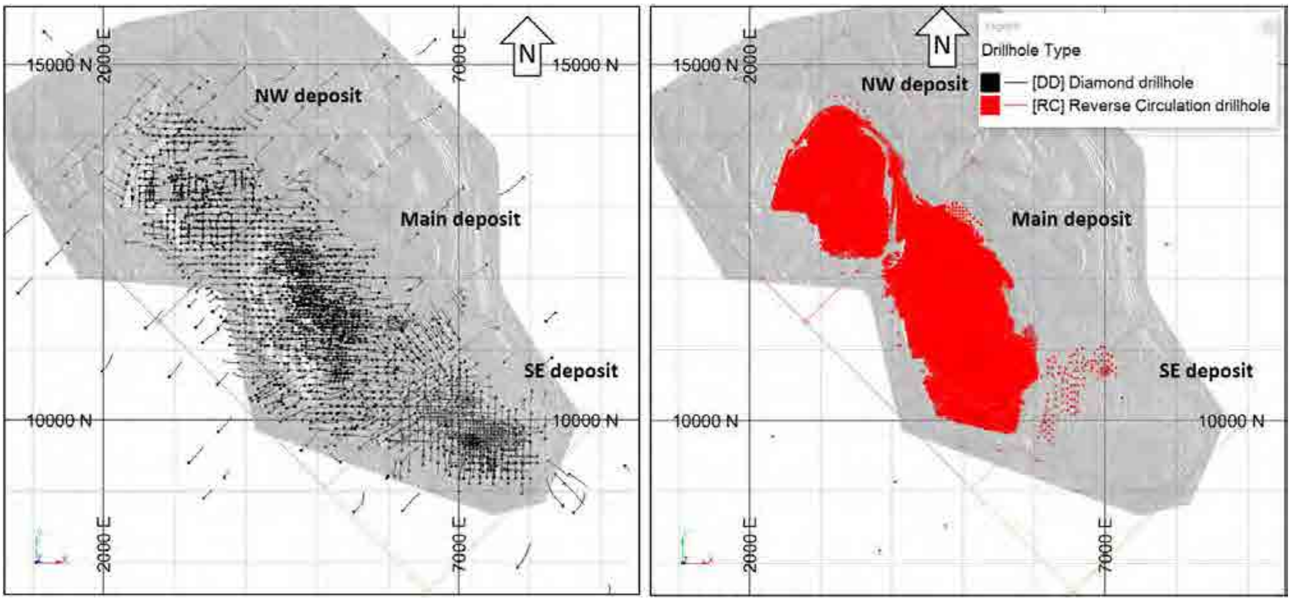
In addition to diamond drilling (DD), reverse circulation (RC) ore control drilling data has also been added across the Main, North West and South East deposits since late 2019. This includes 20,663 additional holes for 1,285,397 additional meters, bringing the total number of RC holes to 75,203 holes and 4,472,329 m. The total number of holes with metres drilled and the number of sample analysis for copper (Cu), acid soluble copper (ASCu) and gold (Au) is summarised in Table 14-1.

Table 14-1 Total number of holes from which data was included in this estimate

Used in 2024 estimate	Drilling	# Holes	# Meters	# Cu samples	# ASCu samples	# Au samples
	DD	2,220	740,709	536,634	492,723	426,286
	RC	75,203	4,472,329	1,770,025	1,769,828	338,270

Figure 14-1 shows the location of available diamond and reverse circulation drilling and the model extents for each deposit.

Figure 14-1 Location of North West, Main and South East deposits at Kansanshi, showing mining extents as at December 2023 and drill hole locations. Diamond drill holes (left); Reverse Circulation (right) (FQM, 2024)



Drill hole data was exported from the SQL database as comma delimited text (csv) files using DataShed's layouts. The collar, survey, assay, geology and dry bulk density csv files were then imported into the Datamine file format. Data validation checks were performed to identify duplicated data, overlapping data, drill holes with no valid collar, or collars with no valid data. Any errors found were addressed by KMP's Database Manager. Additionally, visual inspection of the magnitude of downhole deviation was completed to correct any improperly recorded downhole survey values. Assay data for each element was also interrogated for values outside the expected limits.

The imported Datamine drill hole data was de-surveyed using Datamine's HOLES3D process to create 3D drill hole files of logging and sample assay data. Missing assay data was assigned absent values, indicating either a cavity or lost sample, not waste. Detection limit values for the analytical technique used for each element were recorded as negative values and set to half the detection limit value.

14.4 Geology modelling

KMP geologists generated geological wireframes to define domain boundaries and guide the estimation methods. These included stratigraphic boundaries, veins, intrusive bodies (gabbroic-dioritic), highly weathered volumes, and the mineralisation oxidation profile.

The 3D modelling of the geology was assisted by implicit modelling in Leapfrog software, which created a 3D surface or a 3D solid for each entity. The following geology and wireframe methods were used:

- The original topography (pre-mining) and surveyed pit status were modelled as DTM surfaces, covering the full extent of the KMP area, defining the base of current operations and the upper surface limit.
- Stratigraphy units were modelled as surfaces.
- Intrusive gabbroic/dioritic units, generally barren, were modelled as 3D solids.
- The weathering profile was modelled as surfaces, automatically generated to support the estimated weathering block model volume.
- Highly weathered volumes were modelled as 3D solids.
- Oxidation was modelled as surfaces, categorising areas as refractory (not processable), oxide/transitional and sulphide, to support the estimated oxidation block model volume.

- Vein wireframes were modelled as 3D solids for the North West and Main deposits, and were automatically generated from centre-lines for the South East deposit.

14.4.1 Strata model

The strata wireframes were used to create a block model and populate the fields STRATN and GEOLN with their respective zone values (Table 14-2). An additional clastic unit within the Lower Marble stratum, named Lower Marble Clastics, was also defined and assigned a separate GEOLN code.

Table 14-2 Stratigraphic field values of the model and drill hole sample data

Description	STRATN (Field value)	GEOLN
Upper Pebble Schist (UPS)	199	199
Top Marble (TM)	200	200
Upper Mixed Clastics (UMC)	201	201
Upper Marble (UM)	202	202
Middle Mixed Clastics (MMC)	203	203
Lower Calcareous Schist (LCS)	204	204
Lower Marble (LM)	205	205
Lower Marble Clastics (LMC) [clastic unit within Lower Marble]		255
Lower Pebble Schist (LPS)	206	206
All strata below the LPS surface. Consists of dolerite, schists and phyllite	207	207
Diorite	300	300
Laterite	900	900

14.4.2 Weathering model

Probabilistic volume estimates were included to improve the accuracy of weathering volumes in complex areas and along deeper structures where weathering can be more pervasive. A summary of the defined weathered zones and their respective field WEATHN values is provided in Table 14-3. To support the estimate, especially in areas with sparse drilling data, the logged weathering contact points for the respective weathering horizons (i.e. residual, laterite, saprolite, saprock and fresh rock), were interpolated into a block model and then used to generate 3D DTM surfaces. These surfaces adequately capture the weathering zones in most areas and were used to define initial weathering zones on which the probabilistic estimate was run for local improvement.

Table 14-3 Weathering zone values

Weathering zone	WEATHN
Residual (soil - anthropogenic)	5
Laterite	4
Saprolite	3
Saprock	2
Fresh	1

The probabilistic estimate, using a categorical indicator, applies a semi-horizontal search ellipse for the shallow weathered material and an isotropic search for deeper material below the base of saprock. The isotropic search captures sub-vertical trends, allowing for the accurate prediction of highly weathered material along sub-vertical structures (Figure 14-2 and Figure 14-3).

Figure 14-2 West - East vertical section through the Main deposit showing weathering logged in drill holes and the block modelled weathering volumes (FQM, 2024)

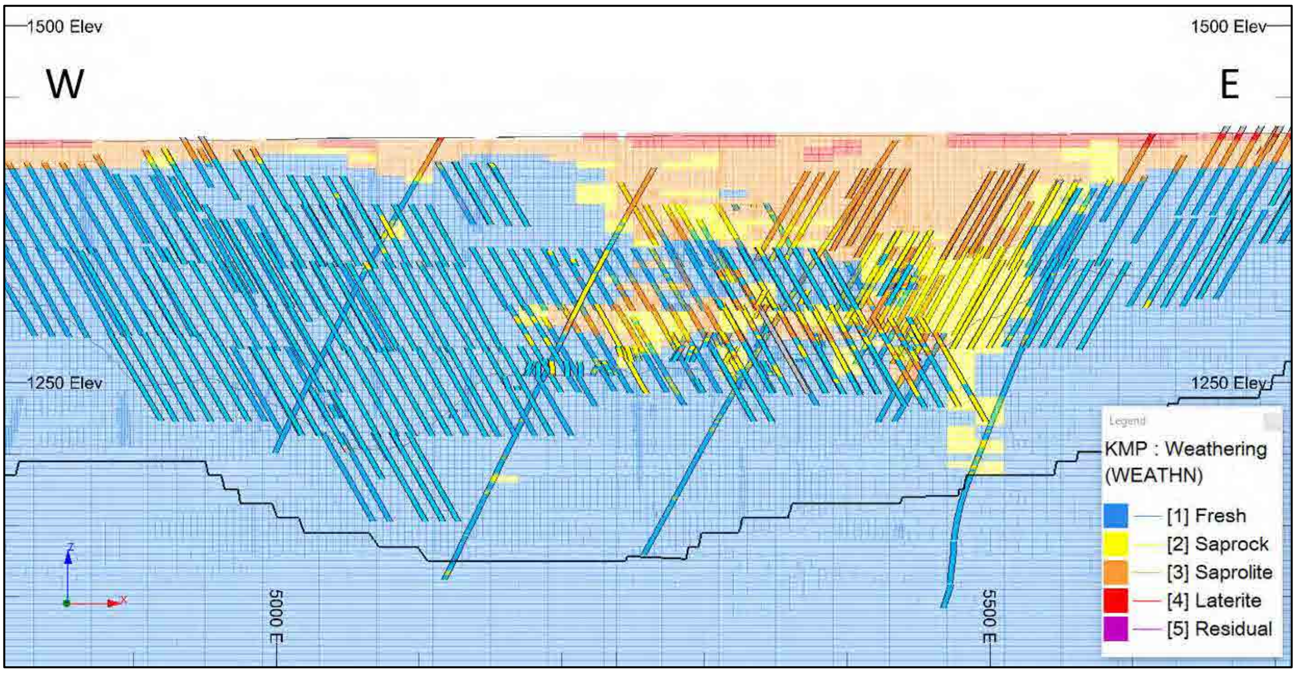
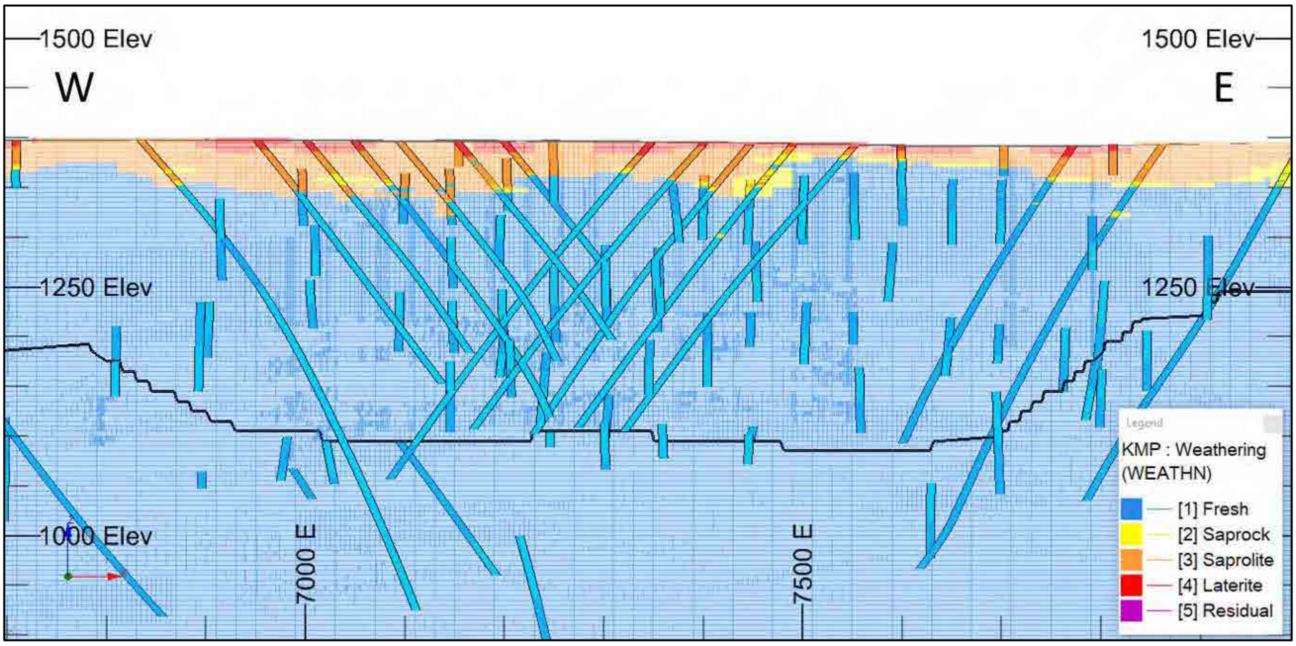


Figure 14-3 West – East vertical section through the South East deposit showing weathering logged in drill holes and the block modelled weathering volumes (FQM, 2024)



14.4.3 Oxidation model

Degrees of oxidation were defined based on the impact of weathering on copper mineralisation (Table 14-4). Similar to weathering, oxidation uses surfaces (i.e. refractory, oxide, mixed and fresh) that were previously interpreted from drill hole geological logging. These interpretations serve as a background for a probabilistic method estimate to better capture the irregular oxidation profile. Drilling data was composited to 5 m lengths, according to oxide coding, to smooth localised variation and better reflect the selectivity at the scale of 5 m mining benches. Samples were coded with their respective oxidation levels, defined by the ratio between acid soluble copper and total copper values (Table 14-4).

Table 14-4 Oxidation field description

OXID	Definition	Classification criteria
10	Fresh	Cu>0.08 AND ASCu/Cu<=0.10
20	Mixed	Cu>0.08 AND ASCu/Cu >0.10 AND ASCu/Cu <=0.5
30	Oxides	Cu>0.08 AND ASCu/Cu >0.5
40	Refractory	Cu>0.08 AND ASCu/Cu <=0.10 AND depth below surface <35 m

Variography (spatial analysis) and search parameters were established for the categorical values associated with each oxide domain. These categorical values were then estimated into the block model, which each block being assigned to the oxide domain with the highest probability value (Figure 14-4 and Figure 14-5).

Figure 14-4 West – East vertical section through the Main deposit showing oxidation in the drill hole intersections and in the block model estimated oxide domains (FQM, 2024)

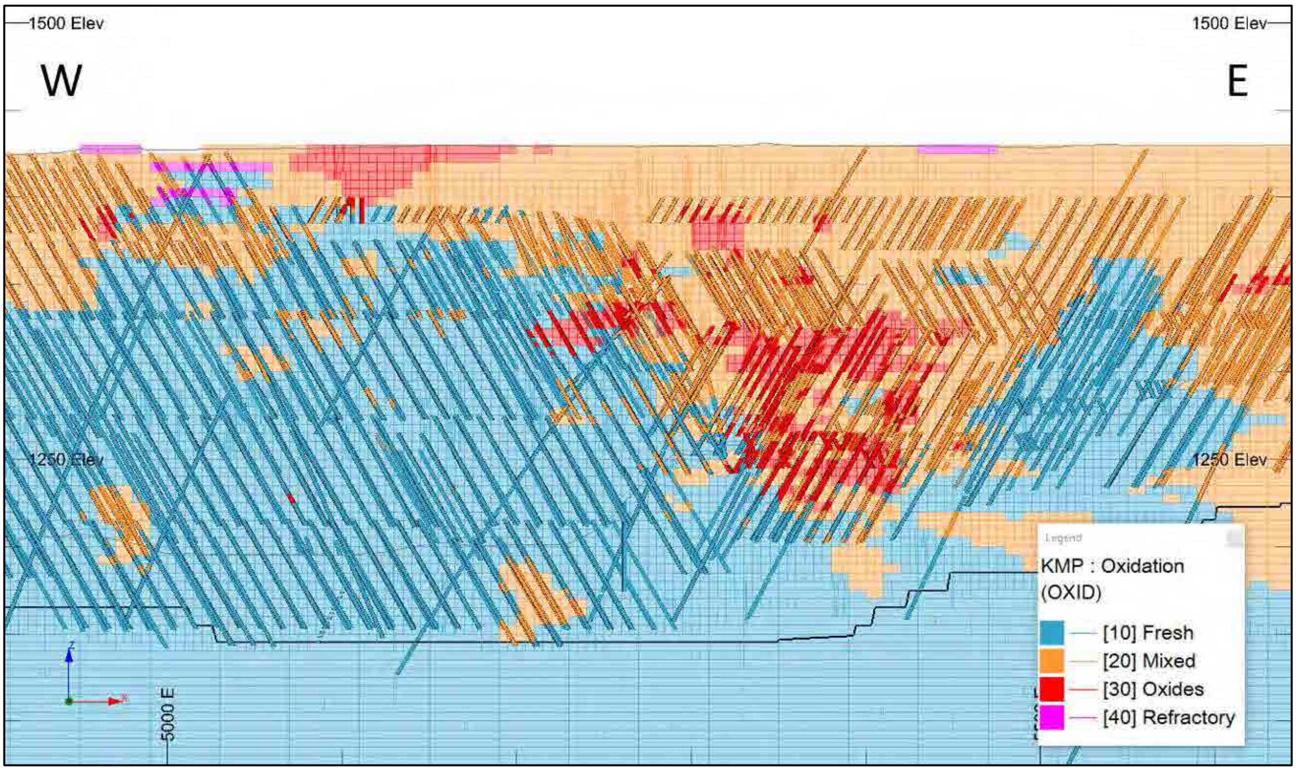
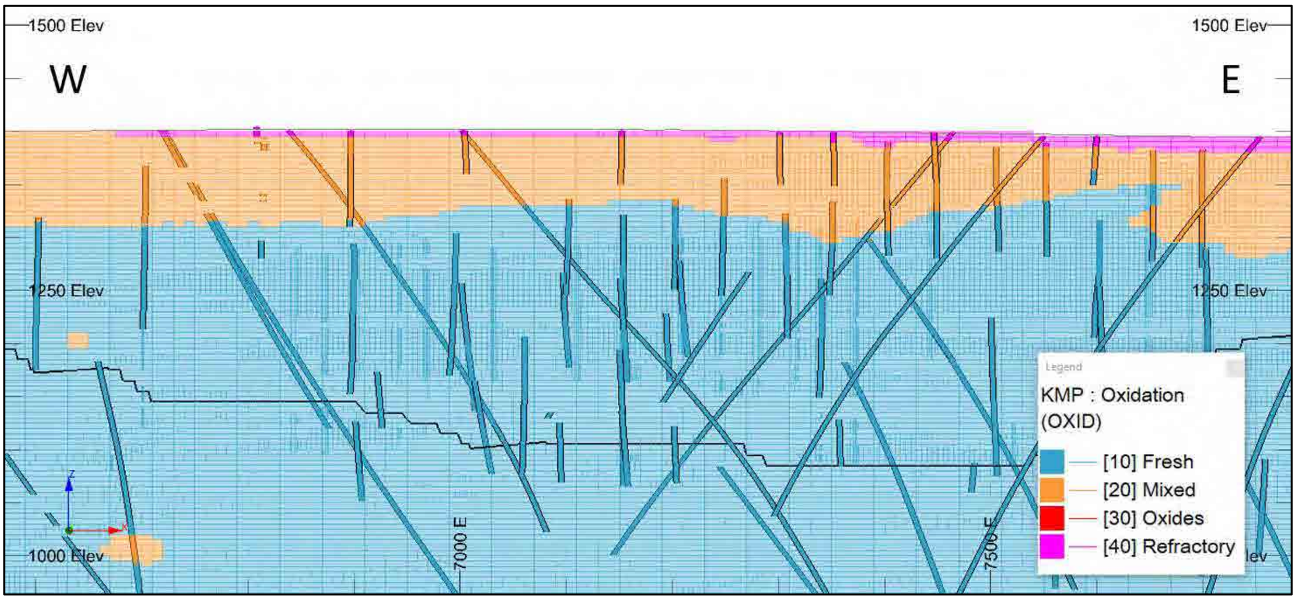


Figure 14-5 West - East vertical section through the South East deposit showing oxidation in the drill hole intersections and in the block model estimated oxide domains (FQM, 2024)



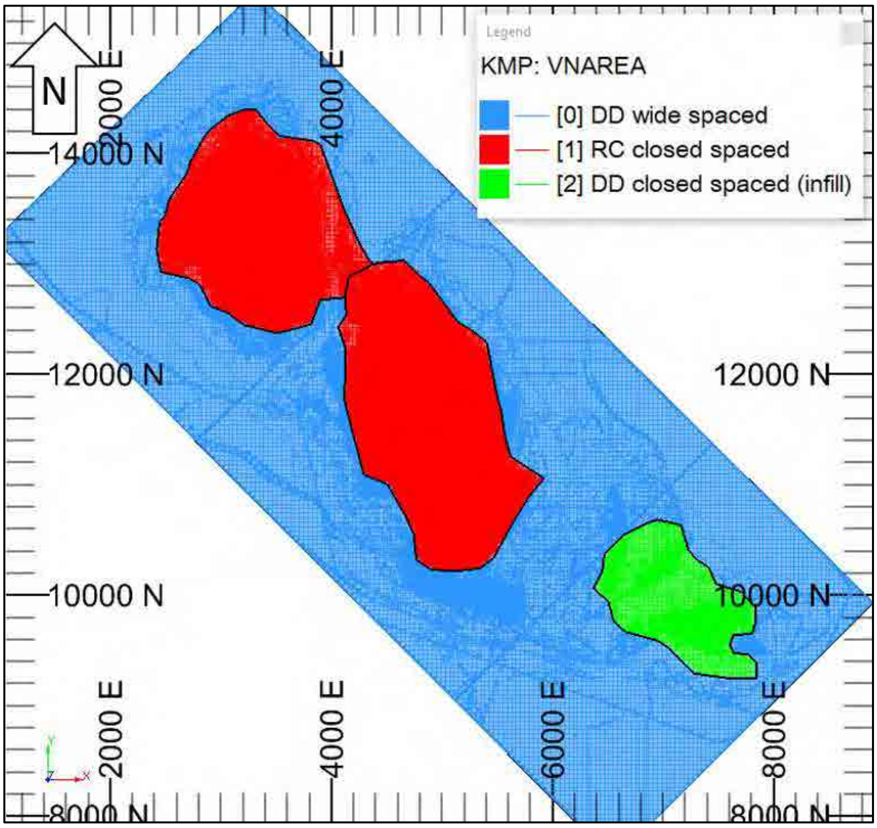
14.4.4 Vein model

Vein modelling methods have continued to evolve, leading to improved definition of vein locations, extents, volumes and attitudes.

Three vein modelling methods were employed based on the available drill hole grid spacing and pit mapping data availability, as illustrated in Figure 14-6:

- For areas designated as VNAREA=0, characterised by wide-spaced drilling (exploration diamond drill holes), vein blocks were flagged based on probability using categorical indicators. The vein thickness for each identified vein block was estimated using true vein thickness data from drilling. Finally, a proportion of vein volume was calculated and assigned to the parent blocks containing vein material.
- For areas designated as VNAREA=1, characterised by close-spaced drilling (ore control drill holes), wireframes were generated in Leapfrog (implicit modelling) with support from pit mapping data, logging data and copper assays.
- For areas designated as VNAREA=2, characterised by close-spaced diamond drilling in the South East Deposit, vein wireframes were created from digitised centrelines using logged vein samples and vein strike data obtained from structural measurements of diamond drill holes.

Figure 14-6 Plan view maps showing the three vein model area (VNAREA) locations (FQM, 2024)



Vein Model in Main and North West deposits (VNAREA 1)

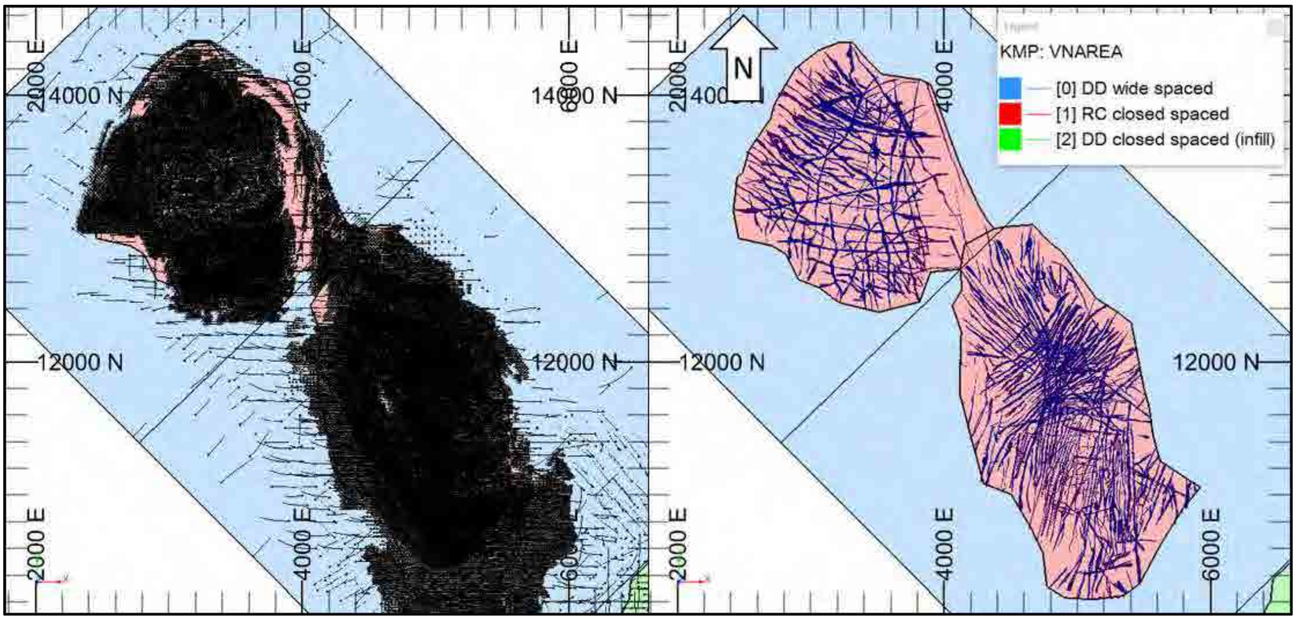
Veins within VNAREA=1 were typically confined to the middle mixed clastics (MMC) stratigraphic unit at the North West deposit and to MMC, upper mixed clastics (UMC) and Lower Marble Clastic (LMC) stratigraphic units at the Main deposit.

However, vein samples and mapping data from pit exposures indicate that within the defined highly weathered and fractured areas, vein mineralisation continues through the typically vein-barren strata (UM, LCS and LMC). The modelling process allowed the 3D modelled veins to populate the block model in these areas and flag those volumes for the vein estimates in the UM, LMC.

Preferred vein orientations, identified from pit mapping, were used to guide wireframe modelling in conjunction with the location of domes or anticlines. These preferred orientations were: north-south, east-west, northeast-southwest, and northwest-southeast (Figure 14-7).

To complement the vein logged samples from diamond and RC drilling, additional samples were flagged based on their potential to have been missed as vein samples, using the scoring criteria outlined in section 14.6. These samples were then used to guide the wireframing process in areas lacking mapping data.

Figure 14-7 Plan view maps showing the VNAREA = 1, drill hole location (left) and vein wireframes (right) (FQM, 2024)

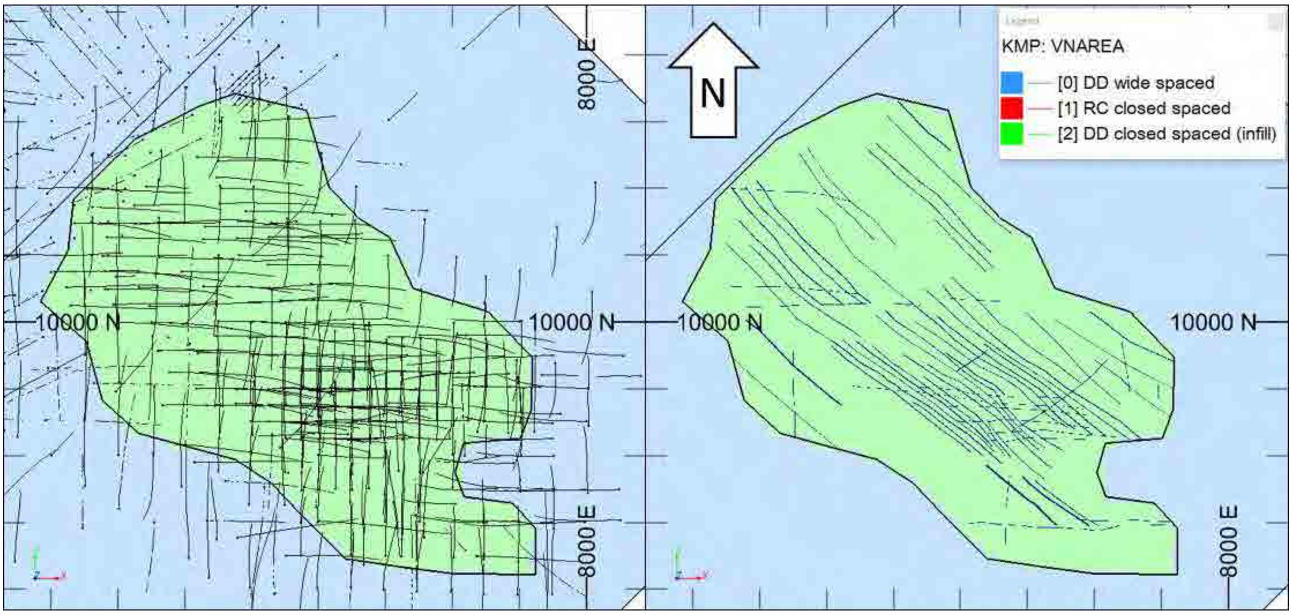


Vein Model in South East deposit (VNAREA 2)

Vein samples from the South East deposit were collected from a closed-space grid of diamond drill holes. For oriented cores, downhole structural measures as vein dip and dip direction were collected and incorporated into the modelling process.

Vein centre-lines were delineated for the UMC, MMC and LM strata, utilising vein strike data for each of the four main orientations (Figure 14-8). Each centreline was assigned a unique vein identifier name, and samples were flagged accordingly. Vein centrelines with fewer than five samples were discarded from further consideration.

Figure 14-8 Plan view maps showing the VNAREA = 2 drill hole location (left) and vein wireframes (right) (FQM, 2024)



Vein true thickness was determined for each sample based on the orientation of the centreline. The average thickness per centreline was then calculated to create string perimeters representing each determined thickness. Veins with true thicknesses lower than one meter were discarded. The remaining vein perimeters were utilised to generate a block model with a resolution up to one meter in both the northing and easting directions, and five meters along the vertical axis. These blocks were flagged with the respective vein identifiers and restricted to the vein hosting strata, namely UMC, MMC and LM. There are no indications of highly weathered areas in the South East deposit that would suggest the presence of vein mineralisation in other strata.

Mining experience from the Main and North West deposits shows an intimate geological and spatial relationship of vein and strata mineralisation, where mineralised veins are predominantly found in mineralised strata and their immediate surroundings. This relationship was applied to the interpretation of vein volumes in the South East deposit, with mineralised strata volumes used to clip the vein volumes prior to conducting the vein grade estimates.

14.5 Sample compositing

Drill hole samples and analytical values were composited down the hole using different intervals based on vein area for both vein and strata data. The selection of the compositing interval utilised the most predominant sample interval lengths for each VNAREA:

- All strata samples were composited into three metre intervals.
- Vein samples for VNAREA=0, where the majority of the drilling assay data comes from one meter diamond core samples, were composited into one metre intervals to preserve the local variability, minimise sample loss, and provide better resolution for the probabilistic method used for vein estimates, with a special focus on vein thicknesses and volume determination.
- For VNAREA=1, where close grid RC data predominates at three metre interval samples, all vein samples were composited into three metre intervals to provide equal support across DD and RC samples.
- In VNAREA=2, with majority of diamond drilling data at one metre intervals, vein samples were composited into one metre intervals, as in VNAREA=0, to maintain the one metre resolution for thickness determination of the interpreted veins. A summary of the intervals used in shown in Table 14-5.

Table 14-5 Composite intervals length used for strata and vein samples across the VNAREA zones

Composite interval (m)	VNAREA		
	0	1	2
Stata	3	3	3
Vein	1	3	1

Overall, mean and median values were not affected by compositing of the strata or vein samples (Table 14-6).

Table 14-6 Composite intervals length used for strata and vein samples across the VNAREA zones

VEIN AREA	Parameter	STRATA		VEIN	
		Raw samples	Composites	Raw samples	Composites
VNAREA = 0	Mean	0.08	0.08	1.33	1.23
	Median	0.01	0.02	0.09	0.09
VNAREA = 1	Mean	0.41	0.37	2.50	2.55
	Median	0.08	0.07	0.86	0.97
VNAREA = 2	Mean	0.13	0.11	2.62	2.63
	Median	0.01	0.02	0.70	0.89

14.6 Sample domain coding

The composited drill hole sample data was coded for stratigraphy, oxidation, weathering, pit, vein area, and deposit using the block model.

While vein wireframes are representative of vein volumes, they may not enclose all logged vein sample data due to the presence of immediately adjacent halo mineralisation and samples associated with smaller veins not captured by the wireframe modelling. Therefore, samples used for estimating grade into vein volumes were selected as follows:

- Diamond vein data was identified by lithology logging (where LITH1=VN) and/or where the logged vein percentage was ≥50%.
- RC grade control vein data was initially identified by lithology logging (LITH=VN or VN=1); however, due to the three-metre sampling interval combined with the challenging nature of chip logging, potential vein samples may have been missed in the logging process and classified as strata samples. To address this, in 2022 the geology team introduced a scoring system based on several parameters to identify potential vein samples overlooked by visual logging and the use of the LITH and VN fields alone. Each sample could score from zero to eight points. Samples with four or more points were considered to assist in guiding the vein 3D modelling process, taking into account geological, alteration, and grade trends. Samples with scores of six or more, which were also flagged with the VEIN_ID of a close 3D modelled vein wireframe, were converted into vein samples and used in the vein estimates. Samples with a score of eight were automatically classified as vein samples and used in the estimates, as they met all criteria. These parameters were:
  - Mineralisation style in the loggin records: If the mineralisation style captured in the logging was of vein or stringer vein type.
  - Mineralisation code: If a sample was logged with its main minerals as copper mineral (oxides, secondary sulphides, primary sulhides, native copper, etc).
  - Vein mineralisation: If the sample was identified as vein mineralisation (calcite, quartz, etc).
  - Copper assay threshold: If the sample exhibited high-grade copper mineralisation (>=0.8% TCu), tipically outside of strata mineralisation.
  - Copper grade contrast: To identify sharp down-the-hole changes in sample copper grades, tipically related to veins. It used a factor of 2.0 to the grade to identify the contrast.
  - Extreme high grades: If samples reported extreme high grades, which are almost exclusively related to vein mineralisation.
  - Spatial location: If the samples were located within or in the inmediate vicinity of a wireframed vein.
- Drilling data not logged as vein was flagged as VEIN=0 and VNGROUP=0 and was therefore used in the stratigraphy grade estimate.

14.7 Estimation domains

The block model was subdivided into areas based on the density of drill hole data, distinguishing between close-spaced RC drilling and wider-spaced diamond drilling. A block model field, SMUDRSCL, was assigned to each block using a nearest neighbourhood estimation method. Blocks with close-spaced RC grade control drilling were assigned an SMUDRSCL value of 1, while blocks not having close-spaced RC drilling, had the SMUDRSCL field value set to zero. The SMUDRSCL field was set using a search ellipse of 30 m by 30 m by 5 m in the X, Y, and Z directions respectively, with a maximum of one sample per drill hole. Blocks with three or more samples, representing three or more drill holes, were allocated a SMUDRSCL value of 1. Grade estimates for each of the final estimation domains were then conducted separately for these two SMUDRSCL areas.

The boundary between copper oxide mineralisation and mixed copper oxides/sulphides (OXID=20 and OXID=30) appears diffuse with irregularly distributed shapes. The oxide and mixed copper domains were merged for estimation purposes.

Mineralised and unmineralised strata were distinguished using a categorical indicator estimate with a 0.08% Cu grade threshold. Probability values of 50% or more were categorised as strata mineralisation. For estimation purposes, the strata was treated as a single unit (combining strata 199 to 206), with the domains separated by oxidation domain. The values of the domain fields are listed in Table 14-7.

Table 14-7 Strata estimation domain fields and values

Description	ESTDOMOW (Field value)
Not estimated e.g. STRATN, 207, 300, 900 or vein mineralisation i.e. VNGROUP>0	0
Primary (sulphide) not mineralised	1099
Primary (sulphide) mineralised	1100
Secondary (oxide) not mineralised	2099
Secondary (oxide) mineralised	2100

The block model volume was subdivided into the three areas corresponding to the levels of supporting data for vein modelling.

- (VNAREA=0) defines the region with wider spaced drilling (resource drilling).
- (VNAREA=1) comprises volumes with available pit mapping data and close-space drilling (ore control)
- (VNAREA=2) comprises volumes for close-space diamond drilling in SED (infilled exploration grid, no ore control drilling)

Historically, veins were grouped by vein area, orientation and deposit using a field called VNGROUP. The Main and North West deposit wireframed veins (VNAREA=1) were coded according to their stratum and respective direction group (Table 14-8). Veins within SED (VNAREA=2) were coded according to their orientation (Table 14-9). Volumes within VNAREA=0 were flagged by deposit and stratigraphic horizon (Table 14-10). This grouping was maintained in this MRE to assists the mine planning and operation teams in grouping vein directions in the context of mining.

Table 14-8 Vein domains for VNAREA=1 (close spaced drilling area) and strata mineralisation volumes

VNAREA	VNGROUP	DEP	STRATA	ORIENTATION	*Description*
ALL	0	ALL	ALL	-	*Strata mineralization across all deposits and VNAREAs*

VNAREA	VNGROUP	DEP	STRATA	ORIENTATION	*Description*
1	1	NWP	MMC	EW	*Wireframed veins in Grade Control area (VNAREA=1) for NWP in MMC strata*
	2			NS	
	3			NWSE	
	4			NESW	
	5	MNP	MMC	EW	*Wireframed veins in Grade Control area (VNAREA=1) for MNP in MMC strata*
	6			NS	
	7			NWSE	
	8			NESW	
	9		UMC	EW	*Wireframed veins in Grade Control area (VNAREA=1) for MNP in UMC strata*
	10			NS	
	11			NWSE	
	12			NESW	
	13		below MMC	EW	*Wireframed veins in Grade Control area (VNAREA=1) for MNP below MMC strata. As modelled, e.g. some veins have been extended to LPS, some others stop at LM, etc*
	14			NS	
	15			NWSE	
	16			NESW	
	17		UM	ALL	*Wireframed mineralised veins & faults are continuous within highly weathered zones. Mineralisation continues through UM and LCS.*
	18		LCS		
	200	MNP & NWP	UMC & MMC & LCS	ALL	*Vein samples not used by vein estimates. Proportion of vein metal into single strata SMUs*

Table 14-9 Vein domains for VNAREA=2 (SED Infilled wide spaced drilling area)

VNAREA	VNGROUP	DEP	STRATA	ORIENTATION	*Description*
2	19	SED	UMC & MMC & LMC	EW	*Interpreted veins in SED (VNAREA=2) from DD vein samples and DD core vein structural data.*
	20			NS	
	21			NWSE	
	22			NESW	

Table 14-10 Vein domains for VNAREA=0 (wide spaced drilling area)

VNAREA	VNGROUP	DEP	STRATA	ORIENTATION	Description
0	30	MNP	UMC	ALL	Probabilistic veins in Resource drilling area (VNAREA=0) for MNP in UMC strata
	40		MMC	ALL	Probabilistic veins in Resource drilling area (VNAREA=0) for MNP in MMC strata
	~~50~~		~~LCS~~	~~ALL~~	Not in use
	60		LM	ALL	Probabilistic veins in Resource drilling area (VNAREA=0) for MNP in LM strata
	70		LPS	ALL	Probabilistic veins in Resource drilling area (VNAREA=0) for MNP in LPS strata
	80	NWP	MMC	ALL	Probabilistic veins in Resource drilling area (VNAREA=0) for NWP in MMC strata
	~~90~~		~~LCS~~	~~ALL~~	Not in use
	~~100~~		~~LM~~	~~ALL~~	Not in use
	~~110~~		~~LPS~~	~~ALL~~	Not in use
	120	SED	UMC	ALL	Probabilistic veins in Resource drilling area (VNAREA=0) for SED in UMC strata
	130		MMC	ALL	Probabilistic veins in Resource drilling area (VNAREA=0) for SED in MMC strata
	~~140~~		~~LCS~~	~~ALL~~	Not in use
	150		LM	ALL	Probabilistic veins in Resource drilling area (VNAREA=0) for SED in LM strata
	~~160~~		~~LPS~~	~~ALL~~	Not in use

The grade estimates of modelled veins in VNAREA 1 and VNAREA 2, utilised an estimation domain called VNESTDOM, which combines three main zone variables: deposit, strata and oxidation. The values of the estimation domain took the form: DEP*100000 + STRATN*100 + OXID, and are detailed in .

Table 14-11 Vein estimation domains (VNESTDOM) values for VNAREA = 1 and VNAREA = 2

Deposit	DEP	Stratigraphy	STRATN	Oxidation	OXID	VNESTDOM
Main	1	UMC	201	Fresh	10	120110
				Mixed-Oxides	20	120120
		UM	202	Fresh	10	120210
				Mixed-Oxides	20	120220
		MMC	203	Fresh	10	120310
				Mixed-Oxides	20	120320
		LCS	204	Fresh	10	120410
				Mixed-Oxides	20	120420
		LM	205	Fresh	10	120510
				Mixed-Oxides	20	120520
		LPS	206	Fresh	10	120610
				Mixed-Oxides	20	120620
North West	2	UMC	201	Fresh	10	220110
				Mixed-Oxides	20	220120
		MMC	203	Fresh	10	220310
				Mixed-Oxides	20	220320
South East	3	UMC	201	Fresh	10	320110
				Mixed-Oxides	20	320120
		MMC	203	Fresh	10	320310
				Mixed-Oxides	20	320320
		LM	205	Fresh	10	320510
				Mixed-Oxides	20	320520

To estimate the gold values in the stratigraphic mineralisation, multiple indicator kriging (MIK) was employed to better handle the strongly skewed distribution of Au grades and prevent excessive top-cutting that could result in the loss of high-grade samples. A gold domain field was created by combining the deposit and mineralised domain, as shown in Table 14-12.

Table 14-12 MIK estimation domains for Au in the strata

Description	ESTDOMIK (Field value)
Not estimated using MIK	0
Main Deposit - Primary (sulphide) mineralised	101100
Main Deposit - Secondary (oxide) mineralised	102100
North West Deposit - Primary (sulphide) mineralised	201100
North West Deposit - Secondary (oxide) mineralised	202100
South East Deposit - Primary (sulphide) mineralised	301100
South East Deposit - Secondary (oxide) mineralised	302100

14.8 Top cuts

Top cuts were implemented to mitigate the impact of high-grade sample values on block estimates. These top cut values were determined by visually inspecting the cumulative distribution frequency curves for each domain. Upon review, the impact of the applied top cuts on the statistics for each domain was found to be negligible.

Top cuts were applied separately to stratiform mineralisation (Table 14-13), wireframed veins in Main and Northwest deposits (Table 14-14), wireframed veins in South East deposit (Table 14-15), and un-wireframed veins in VNAREA=0 (Table 14-16).

Table 14-13 Top cut values applied to strata estimates by deposit and mineralised domain (ESTDOMOW)

Deposit		North West		Main		South East	
ESTDOMOW		1100	2100	1100	2100	1100	2100
Top cut value	Cu%	15.0	15.0	15.0	15.0	15.0	15.0
	ASCu%	1.0	5.0	1.0	5.0	1.0	2.0
	Auppm	0.8	1.5	2.0	2.0	1.0	0.75

Table 14-14 Top cut values applied to vein samples for VNAREA=1 (close space drilling)

Deposit	Strata	Oxidation	VNESTDOM	Top cut value		
				Cu %	**ASCu %**	**Au ppm**
Main	UMC	Fresh	120110	16.0	3.0	2.4
		Mixed - Oxides	120120	37.0	27.5	8.0
	UM	Fresh	120210	17.0	1.75	2.5
		Mixed - Oxides	120220	30.0	23.5	6.3
	MMC	Fresh	120310	28.0	13.0	20.0
		Mixed - Oxides	120320	45.0	23.0	9.5
	LCS	Fresh	120410	20.0	1.3	5.5
		Mixed - Oxides	120420	26.0	14.5	10.0
	LM (LMC)	Fresh	120510	23.0	3.3	6.0
		Mixed - Oxides	120520	27.0	18.3	4.5
	LPS	Fresh	120610	10.8	1.5	6.0
		Mixed - Oxides	120620	9.9	6.3	1.0
North West	UMC	Fresh	220110	-	-	-
		Mixed - Oxides	220120	16.0	12.5	1.2
	MMC	Fresh	220310	33.0	17.0	10.0
		Mixed - Oxides	220320	35.5	31.3	10.0

Table 14-15 Top cut values applied to vein samples for VNAREA=2 (SED)

Deposit	Strata	Oxidation	VNESTDOM	Top cut value		
				Cu %	**ASCu %**	**Au ppm**
South East	UMC	Fresh	320110	22.0	1.8	12.0
		Mixed - Oxides	320120	23.0	11.5	6.0
	MMC	Fresh	320310	22.0	1.0	8.5
		Mixed - Oxides	320320			
	LM (LMC)	Fresh	320510	10.5	0.25	4.0
		Mixed - Oxides	320520			

Table 14-16 Top cut values applied to vein samples for VNAREA=0 (wide space drilling)

Deposit		North West		Main						South East			
Strata		MMC		UMC		MMC		LM	LPS	UMC		MMC	LM
Oxidation		Fresh	Mixed - Oxides	Fresh	Mixed - Oxides	Fresh	Mixed - Oxides	Fresh	Fresh	Fresh	Mixed - Oxides	Fresh	Fresh
Topcut value	Cu%	31.5	35.5	14.0	37.0	28.0	41.0	24.5	12.0	22.0	25.4	23.0	11.0
	AsCu%	17.0	31.0	3.0	14.0	10.5	25.5	3.3	2.5	2.0	13.0	1.1	0.2
	Auppm	12.5	18.5	3.3	7.6	15.0	8.5	8.5	7.0	13.0	6.0	11.0	4.0

No top-cuts were applied for the unmineralised strata domains which had low values and low coefficients of variation (i.e. below 1.5).

14.9 Variography

Spatial analysis of grade continuity (variography) was conducted for Cu, ASCu and Au across each estimation domain, including stratigraphy and vein domains per deposit and oxidation.

Directional variogram analysis utilised a total sill value normalised to the sample population variance within the domains. Nugget variances were determined from downhole variograms at a lag value corresponding to the composite interval length (1 or 3 m). Nugget values were deemed acceptable and ranged from 10% to 35% of the total sill value.

Variogram models for strata domains employed three spherical model structures to effectively define grade continuity. ASCu and Au values were estimated using the same variogram model as Cu.

Variogram models for vein domains in VNAREA=1 utilised two spherical model structures to optimise grade continuity. Domains within the same deposit and stratigraphic unit employed similar variogram models and sample selection support, except for vein domains for MMC at the Main deposit, where only one spherical structure was necessary to fit the experimental variogram, with slightly different sample selection applied to the different vein orientations.

Variogram models for vein domains in VNAREA=2 used one spherical structure with similar nugget, ranges, and sample selection support for the six domains, that are rotated for each vein orientation by using dynamic anisotropy vectors.

For vein estimates in VNAREA=0 (wide-spaced drilling), directional variograms were aligned with the primary vein orientations based on the 3D vein wireframe models per unit and deposit. One spherical model structure was used for each vein direction to fit these experimental variograms.

Variogram models and resulting search parameters used for strata estimation are summarised in Table 14-17 to Table 14-19.

Table 14-17 Summarised variogram models for strata estimates

Domain (ESTDOMOW)	Grade	Variogram rotation angles			Nugget	Spherical model 1 ranges				Spherical model 2 ranges				Spherical model 3 ranges			
		Z	X	Z		X	Y	Z	Sill	X	Y	Z	Sill	X	Y	Z	Sill
Sulphides (1100) & Mix/Oxides (2100)	Cu	0	0	30	0.35	18	12	19	0.45	68	36	22	0.12	167	199	26	0.08
	ASCu	0	0	30	0.35	18	12	19	0.45	68	36	22	0.12	167	199	26	0.08
	Au	0	0	30	0.35	18	12	19	0.45	68	36	22	0.12	167	199	26	0.08

Table 14-18 Summarised search ellipses for strata estimates

Drill spacing area	Grade	Search axis rotation			First Pass search radius			Second Pass radius multiplier	Third Pass radius multiplier
		Z	X	Z	X	Y	Z		
Wide spaced (diamond drill holes) SMUDRSCL=0	Cu	using dynamic anisotropy vectors			80	100	20	1.5	4
	ASCu				80	100	12	1.5	4
	Au				80	100	20	1.5	4

Drill spacing area	Grade	Search axis rotation			First Pass search radius			Second Pass radius multiplier	Third Pass radius multiplier
		Z	X	Z	X	Y	Z		
Close spaced (grade control drill holes) SMUDRSCL=1	Cu	using dynamic anisotropy vectors			80	100	20	1.5	-
	ASCu				80	100	12	1.5	-
	Au				80	100	20	1.5	-

Table 14-19 Summarised search parameters for strata estimates

Drill spacing area	Grade	Search Pass	Min # of comp.	Max # of comp.	Search ellipse	Max # comp. per hole
Wide spaced (diamond drill holes) SMUDRSCL=0	Cu, ASCu, Au	First Pass	8	24	initial	6
		Second Pass	10	24	1.5 x the initial	
		Third Pass	14	70	4 x the initial	
Close spaced (grade control drill holes) SMUDRSCL=1	Cu, ASCu, Au	First Pass	10	28	initial	
		Second Pass	10	28	1.5 x the initial	
		Third Pass	-	-	-	

Variogram models and resulting search parameters used for estimation of veins in the wide-spaced drilling area (VNAREA 0) are summarised in Table 14-20 to Table 14-22.

Table 14-20 Summarised variogram models for vein estimates in the wide spaced drilling area VNAREA=0

Deposit	Strata	Grade	Vein direction	Variogram rotation angles			Nugget	Spherical model 1 ranges			
				Z	X	Z		X	Y	Z	Sill
North West - Main	MMC - UMC	Cu, ASCu, Au	E-W	90	0	0	0.2	20	400	200	0.8
			N-S	0	0	0	0.2	20	400	200	0.8
			NW-SE	-45	0	0	0.2	20	400	200	0.8
			NE-SW	45	0	0	0.2	20	400	200	0.8
South East	MMC - UMC	Cu, ASCu, Au	NW-SE	-45	0	0	0.2	20	400	200	0.8
			N-S	0	0	0	0.2	20	400	200	0.8

Table 14-21 Summarised search ellipse parameters for vein estimates in VNAREA=0

Deposit	Strata	Grade	Vein directions	Search axis rotation			First Pass search radius			Second Pass radius multiplier
				Z	X	Z	X	Y	Z	
North West - Main	MMC - UMC	Cu, ASCu, Au	E-W	90	0	0	20	400	200	0
			N-S	0	0	0	20	400	200	0
			NW-SE	-45	0	0	20	400	200	0
			NE-SW	45	0	0	20	400	200	0
South East	MMC - UMC	Cu, ASCu, Au	NW-SE	-45	0	0	20	400	200	0
			N-S	0	0	0	20	400	200	0

Table 14-22 Summarised search parameters for vein estimates in VNAREA=0

Deposit	Strata	Grade	Vein directions	First Search Pass		Second Search Pass			Max # comp. per hole
				Min # comp.	Max # comp.	Search ellipse	Min # comp.	Max # comp.	
North West - Main	MMC - UMC	Cu, ASCu, Au	E-W	6	24	0 x	-	-	24
			N-S	6	24	0 x	-	-	24
			NW-SE	6	24	0 x	-	-	24
			NE-SW	6	24	0 x	-	-	24
South East	MMC - UMC	Cu, ASCu, Au	E-W	6	24	0 x	-	-	24
			N-S	6	24	0 x	-	-	24

Variogram models and resulting search parameters used for estimation of veins in the close-spaced drilling area (VNAREA 1) are summarised in Table 14-23 to Table 14-25.

Table 14-23 Summarised variogram models for vein estimates in VNAREA=1 (close spaced drilling)

VNESTDOM	Grade	Vein directions	Variogram			Nugget	Spherical model 1				Spherical model 2				Spherical model 3			
			rotation angles				ranges				ranges				ranges			
			Z	X	Z		X	Y	Z	Sill	X	Y	Z	Sill	X	Y	Z	Sill
120110	Cu, ASCu, Au	ALL	0	0	0	0.27	47	47	47	0.36	65	65	65	0.17	384	384	384	0.19
120120	Cu, ASCu, Au	ALL	45	10	170	0.198	124	28	42	0.344	234	113	44	0.248	1136	430	45	0.209
120210	Cu, ASCu, Au	ALL	0	0	0	0.27	47	47	47	0.36	65	65	65	0.17	384	384	384	0.19
120220	Cu, ASCu, Au	ALL	45	10	170	0.198	124	28	42	0.344	234	113	44	0.248	1136	430	45	0.209
120310	Cu, ASCu, Au	ALL	0	0	-55	0.143	16	29	30	0.494	44	70	31	0.176	285	333	32	0.187
120320	Cu, ASCu, Au	ALL	0	0	-130	0.286	28	31	30	0.435	91	116	58	0.121	260	519	65	0.159
120410	Cu, ASCu, Au	ALL	0	0	-55	0.143	16	29	30	0.494	44	70	31	0.176	285	333	32	0.187
120420	Cu, ASCu, Au	ALL	0	0	-130	0.286	28	31	30	0.435	91	116	58	0.121	260	519	65	0.159
120510	Cu, ASCu, Au	ALL	0	5	0	0.354	18	19	26	0.245	40	71	56	0.055	343	910	57	0.346
120520	Cu, ASCu, Au	ALL	-55	20	-10	0.276	44	19	26	0.343	172	59	56	0.18	797	382	57	0.201
120610	Cu, ASCu, Au	ALL	0	0	0	0.2	22	22	22	0.41	60	60	60	0.25	471	471	471	0.15
120620	Cu, ASCu, Au	ALL	0	0	0	0.2	22	22	22	0.41	60	60	60	0.25	471	471	471	0.15
220110	Cu, ASCu, Au	ALL	0	0	0	0.24	55	55	55	0.51	273	273	273	0.12	579	579	579	0.13
220120	Cu, ASCu, Au	ALL	0	0	0	0.24	55	55	55	0.51	273	273	273	0.12	579	579	579	0.13

VNESTDOM	Grade	Vein directions	Variogram rotation angles			Nugget	Spherical model 1 ranges				Spherical model 2 ranges				Spherical model 3 ranges			
			Z	X	Z		X	Y	Z	Sill	X	Y	Z	Sill	X	Y	Z	Sill
220310	Cu, ASCu, Au	ALL	0	0	10	0.258	15	32	27	0.498	410	396	56	0.244	-	-	-	-
220320	Cu, ASCu, Au	ALL	0	0	-20	0.17	36	25	19	0.584	711	445	48	0.246	-	-	-	-

Table 14-24 Summarised search ellipses parameters for vein estimates in VNAREA=1

VNESTDOM	Grade	Vein directions	Search axis rotation			First Pass search radius			Second Pass radius multiplier
			Z	X	Z	X	Y	Z	
120110	Cu, ASCu, Au	ALL	using dynamic anisotropy vectors			95	95	8	3
120120						190	70	8	3
120210						95	95	8	3
120220						190	70	8	3
120310						85	70	8	3
120320						130	65	8	3
120410						85	70	8	3
120420						130	65	8	3
120510						150	55	8	3
120520						135	65	8	3
120610						120	120	8	3
120620						120	120	8	3
220110						145	145	8	3
220120						145	145	8	3
220310						105	100	8	3
220320						140	90	8	3

Table 14-25 Summarised search parameters for vein estimates in VNAREA=1

VNESTDOM	Grade	Vein directions	First Search Pass		Second Search Pass			Max # comp. per hole
			Min # comp.	Max # comp.	Search ellipse	Min # comp.	Max # comp.	
120110	Cu, ASCu, Au	ALL	8	20	3 x	8	20	4
120120			8	20	3 x	8	20	4
120210			8	20	3 x	8	20	4
120220			8	20	3 x	8	20	4
120310			8	20	3 x	8	20	4
120320			8	20	3 x	8	20	4
120410			8	20	3 x	8	20	4
120420			8	20	3 x	8	20	4
120510			8	20	3 x	8	20	4
120520			8	20	3 x	8	20	4
120610			8	20	3 x	8	20	4
120620			8	20	3 x	8	20	4

VNESTDOM	Grade	Vein directions	First Search Pass		Second Search Pass			Max # comp. per hole
			Min # comp.	Max # comp.	Search ellipse	Min # comp.	Max # comp.	
220110			8	20	3 x	8	20	4
220120			8	20	3 x	8	20	4
220310			8	20	3 x	8	20	4
220320			8	20	3 x	8	20	4

Variogram models and resulting search parameters used for estimation of veins in the close-spaced diamond drilling area in the South East deposit (VNAREA 2) are summarised in Table 14-26 to Table 14-28.

Table 14-26 Summarised variogram models for vein estimates in VNAREA 2

VNESTDOM	Grade	Vein directions	Variogram rotation angles			Nugget	Spherical model 1 ranges			
			Z	X	Z		X	Y	Z	Sill
320110	Cu, ASCu, Au	ALL	0	0	0	0.2	300	300	50	0.8
320120	Cu, ASCu, Au	ALL	0	0	0	0.2	300	300	50	0.8
320310	Cu, ASCu, Au	ALL	0	0	0	0.2	300	300	50	0.8
320320	Cu, ASCu, Au	ALL	0	0	0	0.2	300	300	50	0.8
320510	Cu, ASCu, Au	ALL	0	0	0	0.2	300	300	50	0.8
320520	Cu, ASCu, Au	ALL	0	0	0	0.2	300	300	50	0.8

Table 14-27 Summarised search ellipses for vein estimates in VNAREA 2

VNESTDOM	Grade	Vein directions	Search axis rotation			First Pass search radius			Second Pass radius multiplier
			Z	X	Z	X	Y	Z	
320110	Cu, ASCu, Au	ALL	using dynamic anisotropy vectors			150	50	10	2
320120						150	50	10	2
320310						150	50	10	2
320320						150	50	10	2
320510						150	50	10	2
320520						150	50	10	2

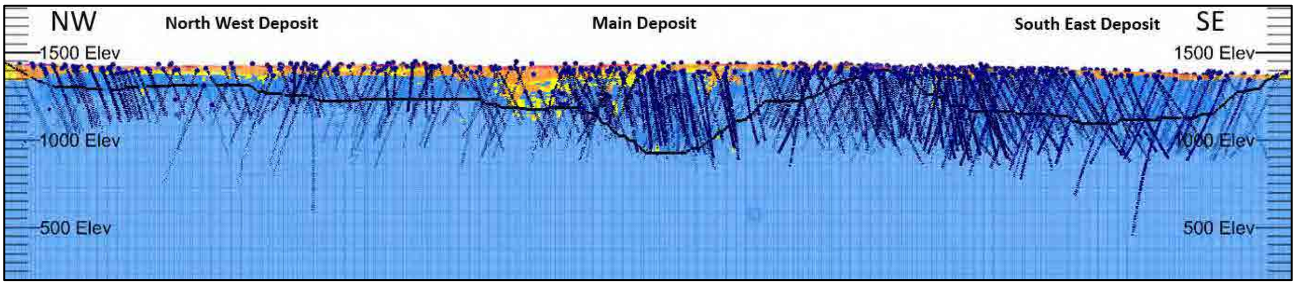
Table 14-28 Summarised search parameters for vein estimates in VNAREA 2

VNESTDOM	Grade	Vein directions	First Search Pass		Second Search Pass			Max # comp. per hole
			Min # comp.	Max # comp.	Search ellipse	Min # comp.	Max # comp.	
320110	Cu, ASCu, Au	ALL	8	24	2 x	6	20	16
320120			8	24	2 x	6	20	16
320310			8	24	2 x	6	20	16
320320			8	24	2 x	6	20	16
320510			8	24	2 x	6	20	16
320520			8	24	2 x	6	20	16

14.10 Dry bulk density

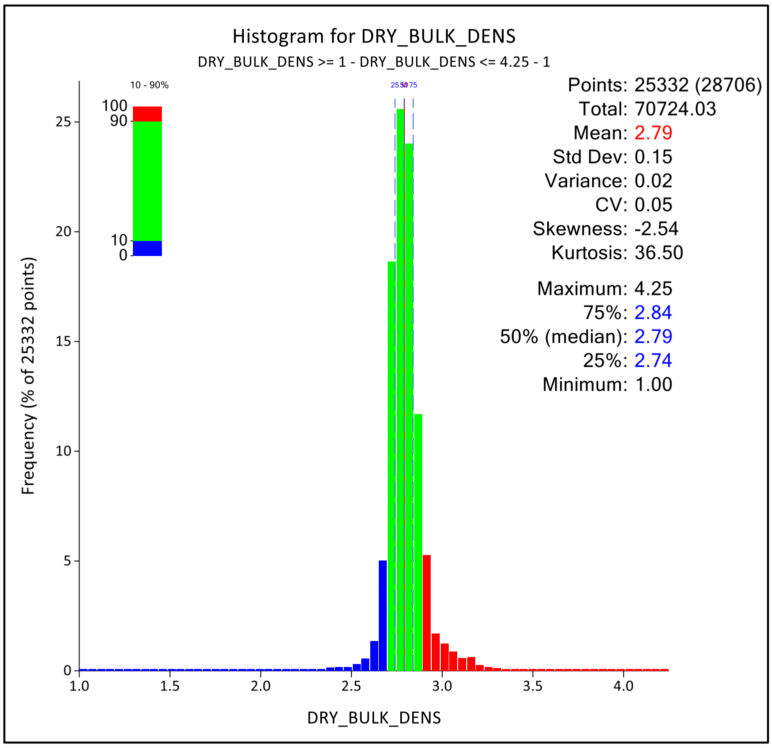
In-situ dry bulk density measurements were obtained for 30,031 samples, offering extensive coverage across the deposits (Figure 14-9). Sample density values were determined using the traditional Archimedes method. Density data underwent statistical analysis, revealing that strata and weathering exerted the most significant influence. The majority of the Kansanshi deposits consist of fresh material with moderate variability in density (Figure 14-10).

Figure 14-9 Dry bulk density sample distribution (in blue) across deposit and weathering horizons (FQM, 2024)



Overall bottom and top cuts were applied at 1.0 and 4.25 g/cm^3 respectively.

Figure 14-10 Histogram for dry bulk density samples in the fresh weathering domain



The estimation domain for dry bulk density (DENSDOM) was determined based on a combination of strata and weathering intensity, calculated using the formula DENSDOM = STRATN*10 + WEATHN. Both, block model and dry bulk density samples were coded accordingly. Dry bulk density was estimated using ordinary kriging with a parent block size of 40 m by 40 m by 5 m. Blocks that were not estimated were assigned the average density values derived from samples within each domain. Table 14-29 provides a summary the average dry bulk density data from samples categorised by weathering domain.

Table 14-29 Dry bulk density mean values by weathering domain and strata

WEATHN	Description	STRATN	DENSITY
1	Fresh	all	2.80
1	Fresh	198	2.89
		199	2.84
		200	2.78
		201	2.80
		202	2.75
		203	2.81
		204	2.78
		205	2.76
		206	2.81
		207	2.84
		300	2.91
		900	2.80

WEATHN	Description	STRATN	DENSITY
2	Saprock	all	2.73
2	Saprock	198	2.73
		199	2.73
		200	2.76
		201	2.71
		202	2.71
		203	2.74
		204	2.74
		205	2.72
		206	2.76
		207	2.80
		300	2.90
		900	2.73

WEATHN	Description	STRATN	DENSITY
3	Saprolite	all	1.63
3	Saprolite	198	1.79
		199	1.70
		200	1.70
		201	1.59
		202	1.49
		203	1.63
		204	1.63
		205	1.63
		206	1.63
		207	1.63
		300	1.68
		900	1.63

WEATHN	Description	STRATN	DENSITY
4	Laterite	all	1.57
5	Residual	all	1.55
ALL	ALL	ALL	2.71

14.11 Block model parameters

For the estimate, five parent block model grids (prototypes), were utilised. The base grid was employed for weathering and oxidation probabilistic estimation methods (categorical indicators), dry bulk density, strata and veins in the wide-spaced drilling areas (VNAREA=0). It had a parent block dimension of 40 m by 40 m by 5 m, sub-celled down to 5 m by 5 m by 5 m. Specific details for each grid are presented in Table 14-30 and Table 14-31.

The close grid area in the strata (SMUDRSCL=1) used a parent block of 5 m by 5 m by 5 m for the estimates. The Localised Uniform Conditioning applied to the strata estimates in the wide grid area (SMUDRSCL=0) utilised the same block size of 5 m by 5 m by 5 m.

The VNAREA=1 vein grid in Main and North West deposits employed a parent block of 20 m by 20 m by 10 m. Meanwhile, the VNAREA=2 vein grid in South East deposit utilized a parent block of 20 m by 20 m by 20 m. Both vein models were sub celled down to 1 m by 1 m by 5 m to more accurately represent the 3D modelled vein volumes.

Upon estimation of all models, they were merged and regularised into the final SMU grid size of 10 m by 10 m by 5 m.

The upper limits of the block model were constrained to the original topographic DTM surface. Any material (blocks) located between the original topography and the pit surveyed DTM, dated 31st December 2023, were flagged and excluded as mined material.

Table 14-30 Kansanshi's parent block model origin and extension coordinates

Block model origin			Block model limit		
X Origin	**Y Origin**	**Z Origin**	**X End**	**Y End**	**Z End**
630 mE	7500 mN	200 mRL	9070 mE	15660 mN	1700 mRL

Table 14-31 Kansanshi's parent block model grids used in this estimate

		Parent cell size			Minimum cell size			Number of parent blocks		
		X	Y	Z	X	Y	Z	X	Y	Z
Weathering model		40 m	40 m	5 m	5 m	5 m	5 m	211	204	300
Oxidation model		40 m	40 m	5 m	5 m	5 m	5 m	211	204	300
Density model		40 m	40 m	5 m	5 m	5 m	5 m	211	204	300
Strata grade model	SMUDRSCL 0	40 m	40 m	5 m	5 m	5 m	5 m	211	204	300
	SMUDRSCL 0 LUC	5 m	5 m	5 m	-	-	-	1688	1632	300
	SMUDRSCL 1	5 m	5 m	5 m	5 m	5 m	5 m	1688	1632	300
	AU MIK (SMUDRSCL 0)	40 m	40 m	5 m	5 m	5 m	5 m	211	204	300
	AU MIK (SMUDRSCL 1)	10 m	10 m	5 m	5 m	5 m	5 m	844	816	300
Vein grade model	VNAREA 0 (Main and NW)	40 m	40 m	5 m	10 m	10 m	5 m	211	204	300
	VNAREA 0 (South East)	40 m	40 m	5 m	5 m	5 m	5 m	211	204	300
	VNAREA 1	20 m	20 m	10 m	1 m	1 m	5 m	422	408	150
	VNAREA 2	20 m	20 m	20 m	1 m	1 m	5 m	422	408	75
SMU regularised model		10 m	10 m	5 m	-	-	-	844	816	300

14.12 Grade estimate – stratigraphy

Estimates for the wide-spaced drilled areas (SMUDRSCL=0) used a parent block dimension of 40 m by 40 m by 5 m, whereas for the closed-spaced areas (SMUDRSCL=1), the estimates utilised a block dimension of 5 m by 5 m by 5 m.

Dynamic anisotropy true dip and dip directions were determined from the true dip and dip direction of the strata wireframes to orient search ellipses and variogram models. These parameters were estimated into each block per strata horizon (STRATN =199 to STRATN=206) to support the ordinary kriging routine.

Sample data used for strata estimation was coded per stratigraphic horizon (STRATN = 199 to 206), with vein samples excluded from the strata estimates.

Cu and ASCu grades were estimated into strata blocks using search ellipses and variogram models oriented to the local dip and strike of strata, i.e. dynamic anisotropy.

Grades were estimated by ordinary kriging into the parent block for the wide spaced DD drilled areas as well as the parent block for the close spaced RC drilled areas. Both sets of parent block estimates were validated against the input sample data through visual inspection, data means and swath slice plots. This validation ensured that parent estimates correlated with input data and had no evident estimation issues.

For the wide-spaced DD drilled areas, validated ordinary kriging parent estimates underwent post-processing using localised uniform conditioning (LUC). LUC is a geostatistical technique used to assess recoverable resources within a parent block by estimating the proportions of selective mining units (SMUs) above a cut-off grade and their corresponding average grade. LUC effectively represents the grade variability within a larger parent block at the scale of mining. SMU dimensions were set to 5 m by 5 m by 5 m.

In close-spaced RC drilled areas, estimates were made directly into SMU-sized blocks, without LUC post-processing. Similarly, the peripheries of wide-spaced DD drilled areas, which often had insufficient samples for reasonable LUC results, relied on kriged or allocated mean values for grade estimation. Allocated mean grade values were determined from top cut and declustered mean sample grades. Finally, the models for both close and wide-grid spacing were combined to produce the final strata model.

14.13 Grade estimate – stratigraphy – Au MIK estimate

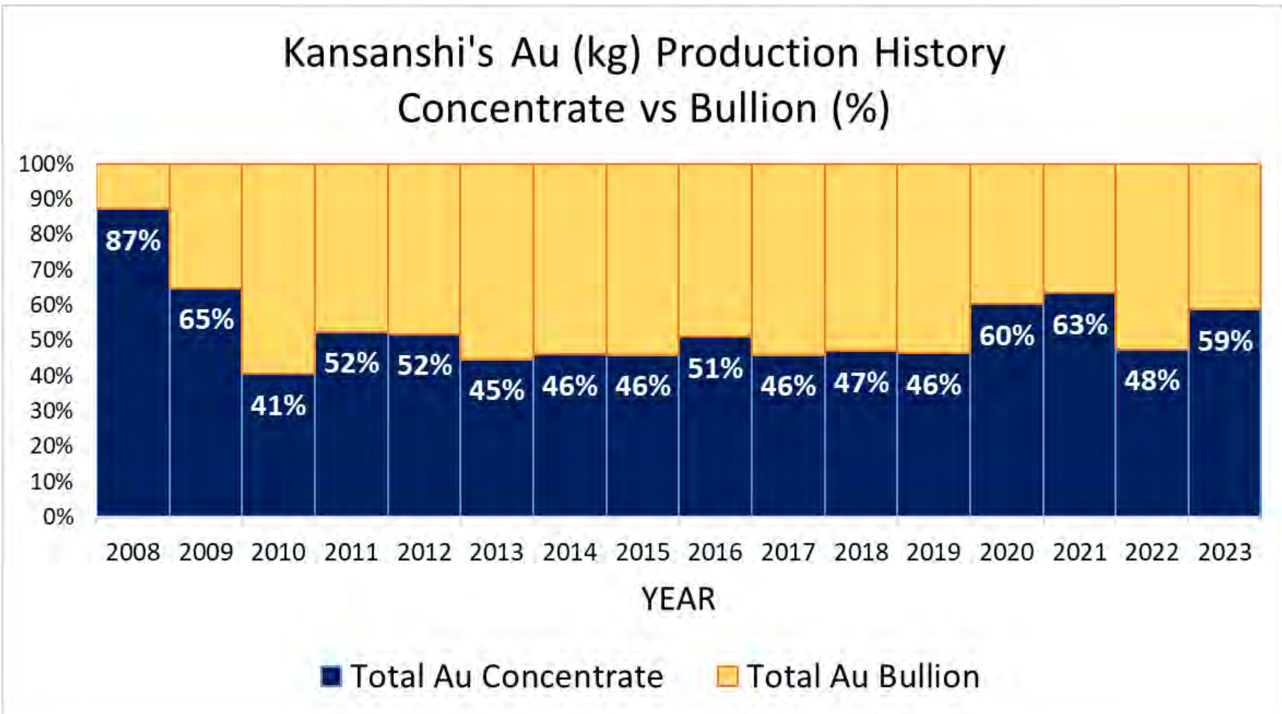
Gold mineralisation at the three Kansanshi deposits is sporadic in nature. Initially, only a small percentage (~10%) of samples showed gold values above analytical detection limits, leading to the belief that gold mineralisation was minor. However, as production progressed and the processing plant improved, FQM recognised an opportunity to recover free gold particles through gravity methods, in addition to the gold already recovered in the concentrate linked to copper mineralisation.

As the gold opportunity increased, the sampling grid intensified. However, both old and new sampling grids revealed a precision error attributed to the nugget effect of gold, particularly with free gold inconsistently captured in samples. To address this bias, larger and more frequent samples were collected, resulting in a slight improvement in precision. Yet, given the nature of free gold mineralisation, achieving complete capture would demand impractically large sample masses and closer drill grids. As a result, the proportion of free gold's contribution to the total gold grade often goes unrecorded in the samples. Mineral Resource estimates have therefore underestimated recoverable gold by approximately 50%, presenting a low operational risk.

Notably, free gold is not considered in the Mineral Reserves, nor the Mineral Resources estimate.

Over the past 16 years, gold produced in concentrates accounted for roughly half of total gold production. Gold represents ~20% of Kansanshi's NPV, with ~10% from free gold captured in the gravity concentrator's line and ~10% from copper-associated gold recovered in the final copper concentrates (Figure 14-11).

Figure 14-11 Kansanshi's gold production history by recovery process



The Mineral Resource estimate classification for gold mineralisation aligns with that of copper mineralisation, which is the main driver for the Mineral Reserves and mine plans.

To address the strongly skewed Au grade distributions, strata Au grades were estimated using multiple indicator kriging (MIK). MIK estimates were completed separately for each deposit in order to limit variations in the cumulative distribution from which the indicator cut-off grades are based. The three deposit areas have different tenors, with the South East deposit being the smallest and having lowest Au grades.

Gold threshold values and indicators (0/1 values) were estimated into the block model volume using Datamine software. The estimated threshold values were processed using the GSLIB POSTIK program, and final gold grades were compiled for each parent cell outline. The final MIK product was an e-type estimate (the average grade of the block) which was then imported back into Datamine. The MIK estimation parameters used are detailed in Table 14-32.

Table 14-32 MIK parameters by deposit and oxidation domain

Deposit		Main		North West		South East	
Oxidation		Sulphide	Mix/Oxides	Sulphide	Mix/Oxides	Sulphide	Mix/Oxides
min. value		0.001	0.001	0.01	0.001	0.001	0.01
max. value		20.0	25.0	9.0	25.0	9.0	5.0
lower tail*		0.9	0.93	0.94	1.0	1.0	1.0
upper tail**		1.71	1.28	2.03	1.52	1.7	2.64
Bins		Au thresholds		Au thresholds		Au thresholds	
Percentile	20	0.01	-	0.008	-	-	-
	30	0.018	0.008	0.01	0.008	0.01	0.007
	40	0.026	0.01	0.02	0.01	0.014	0.009
	50	0.039	0.016	0.03	0.02	0.02	0.011
	60	0.056	0.023	0.04	0.03	0.029	0.015
	70	0.08	0.04	0.064	0.045	0.045	0.022
	80	0.13	0.072	0.111	0.09	0.072	0.042
	90	0.239	0.166	0.25	0.244	0.139	0.104
	95	0.405	0.322	0.45	0.512	0.229	0.228
	97.5	0.683	0.59	0.828	0.96	0.367	0.374
	99	1.279	1.305	1.61	2.031	0.664	0.924
	** power model*						
	*** hyperbolic model*						

14.14 Grade estimate – wireframed veins in Main and North West deposits (VNAREA = 1) and South East (VNAREA = 2)

For block model volumes in VNAREAs 1 and 2, individual sets of veins were modelled and coded for the fields DEP, STRATN and OXID, creating the domain field VNESTDOM. Oxide and mixed vein material were estimated separately from primary sulphide, strata and deposit to limit grade smearing. Boundary analysis supports the use of these respective soft and hard boundaries during estimation.

Vein wireframes set the dynamic anisotropy true dip and dip directions in the vein volume model. The dynamic anisotropy values were estimated into the vein models using inverse distance estimation.

Samples not logged as VEIN were removed. Remaining vein samples were top-cut to reduce the coefficient of variation (ratio of the standard deviation to the mean) to between 1.5 and 2, while minimising the percentage of data being cut. Top-cut values were guided by disintegration of the cumulative distribution plots as per section 14.8.

Any un-estimated blocks were allocated mean grades based on the declustered drillhole statistics for each estimation domain (VNESTDOM).

Individual veins were assigned VNGROUP codes as per their strata and orientation.

14.15 Grade estimate – non-wireframed vein areas (VNAREA 0)

The wide spaced DD areas (VNAREA=0) do not have wireframes to constrain vein grade estimates and instead used the VNGROUP domain. The estimation process first defined the most likely position, volume, and orientation of vein mineralisation, followed by the estimation of their grades.

The mixed-oxide mineralisation was estimated separately from the primary sulphide components using hard boundaries. Boundary analysis confirmed the use of these hard boundary conditions during estimation for these two vein populations.

Vein occurrences across the three deposits at Kansanshi tend to favour sub-vertical orientations. Consequently, vein heights were limited to the vertical extents of the stratigraphic units that have shown a reasonable number of vein intersects in the diamond drilling database, i.e., UMC and MMC for North West deposit; UMC, MMC, LM (LMC) and LPS stratigraphic units for Main deposit; and UMC, MMC and LM (LMC) for South East deposit. The parent (40 m by 40 m by 5 m) block model was re-blocked into a single vertical 40 m by 40 m block according to the height the selected stratigraphic horizons. One metre composited vein samples were coded with an indicator value set to 1, while all non-vein samples were set to 0.

The resulting categorical indicator estimates reflect the probability of the single vertical vein block containing vein mineralisation. Higher estimated values suggest a stronger probability that the block has vein mineralisation. The estimated probability cut-off values used to flag blocks with vein mineralisation were based on the analysis of the estimated probability value distribution per domain and visual correlation with the DD vein logged samples. Parent blocks not flagged as having vein mineralisation were assumed to contain limited to no vein mineralisation.

Vein true thickness estimates were completed for each parent block with vein mineralisation. The method is summarised as follows:

- Vein drill hole samples were composited to single downhole intercept lengths per vein to capture the full length of continuous vein intersections.
- The true thickness of each composited vein interval was calculated for the four main vein strike orientations: north-south, east-west, north-east and north-west. The South East deposit was limited to north-west and north-south vein orientations.
- A categorical indicator estimate was completed for each vein orientation to identify the dominant orientation in each parent block.
- Once the most probable orientation was selected, vein true thickness values for each orientation were estimated into the parent blocks containing vein mineralisation.
- The estimated vein thickness and orientation were used to calculate the proportion of vein volume in each parent block, thus determining the tonnage of the vein mineralisation.

Cu, ASCu and Au grades were estimated into the vein mineralised 40 m by 40 m by 5 m parent blocks from the one metre vein composite samples using ordinary kriging.

Vein grade estimates and assigned vein volumes were adjusted into a pseudo vein block model where vein mineralisation was represented as vertical columns of 10 m by 10 m (Main and North West) blocks or 5 m by 5 m (South East) blocks for the full height of the strata units. The vein grade assigned to these pseudo vein blocks was calculated by dividing the estimated vein tonnes by the corresponding metal, based on the estimated volume proportions. The pseudo vein model's location and shape are representative of vein mineralisation, but it is not locally accurate.

The selected 5 m by 5 m by 5 m South East deposit blocks represents around 1.6% of a 40 m by 40 m by 5 m parent block. This proportion aligns with the average estimated vein proportions for the South East area. The Main and North West deposits had estimated vein proportions two to four times higher than the South East deposit, and therefore a 10 m by 10 m block area was used, which represents a 6.3% volume of a parent block.

In relatively sparsely drilled areas, the use of pseudo vein models effectively reflects the contained vein metal and grade without distorting grade tonnage curve data with step artefacts.

14.16 Mining metal risk

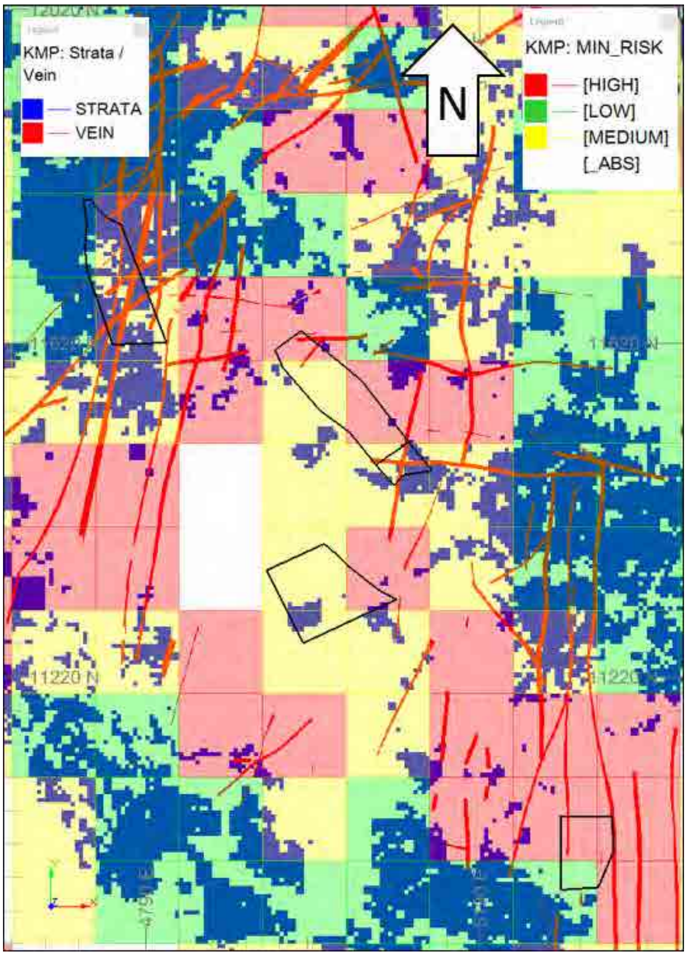
The Mining Metal Risk parameter (MIN_RISK) was introduced in 2023 to help the mine planning and operations teams identify areas of the Mineral Resources and Reserves that have a higher spatial risk for dilution during mining activities such as blasting, loading, hauling, feeding and stockpiling. This parameter utilised panel blocks of 100m by 100m by 5m, which align with volumes of regular blasting units. Within each panel block, the ratio of overall mineralisation (above 0.2% Cu) was calculated, along with the ratio of strata versus vein metal contained.

These values were combined into a risk matrix (Table 14-33). Low-risk panel blocks typically had a high percentage of their volume containing mineralisation, predominantly from strata or strata-dominant source. High-risk panel blocks usually had a low overall ratio of mineralisation and dominance of vein material source over strata, indicating a higher chance for waste dilution during blasting, loading, and other operations (Figure 14-12).

Table 14-33 Mining Metal Risk matrix

MINING METAL RISK		Strata / Vein metal domains				
		Strata only	Strata dominant	Strata and Vein	Vein dominant	Vein only
MINERALISATION VOLUMEN %	<25%	MEDIUM	MEDIUM	HIGH	HIGH	HIGH
	25%-50%	LOW	MEDIUM	MEDIUM	HIGH	HIGH
	50%-75%	LOW	LOW	MEDIUM	MEDIUM	HIGH
	>75%	LOW	LOW	LOW	MEDIUM	MEDIUM

Figure 14-12 Horizontal section in Main deposit showing Mining Metal Risk in the block model, together with strata mineralisation (blue blocks) and vein mineralisation (red blocks). Black squares are examples of different blast perimeters (FQM, 2024)



The Mining Metal Risk matrix was also utilised in the Reserves conversion process to provide spatial context for the modifying factors applied.

14.17 Block model validation

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

- Wireframed vein volumes were compared with block model volumes.
- Cross-sections of sample grade data were viewed against block model grade estimates.
- Mean sample data grades for each domain were compared with mean estimated grades in the block model.
- Swath/trend plots were reviewed along northings, eastings and RL.
- Reconciliation of annual production data.

The wireframes and block model volumes compared well, with a difference of within 1%, attributed to minor resolution loss along block edges.

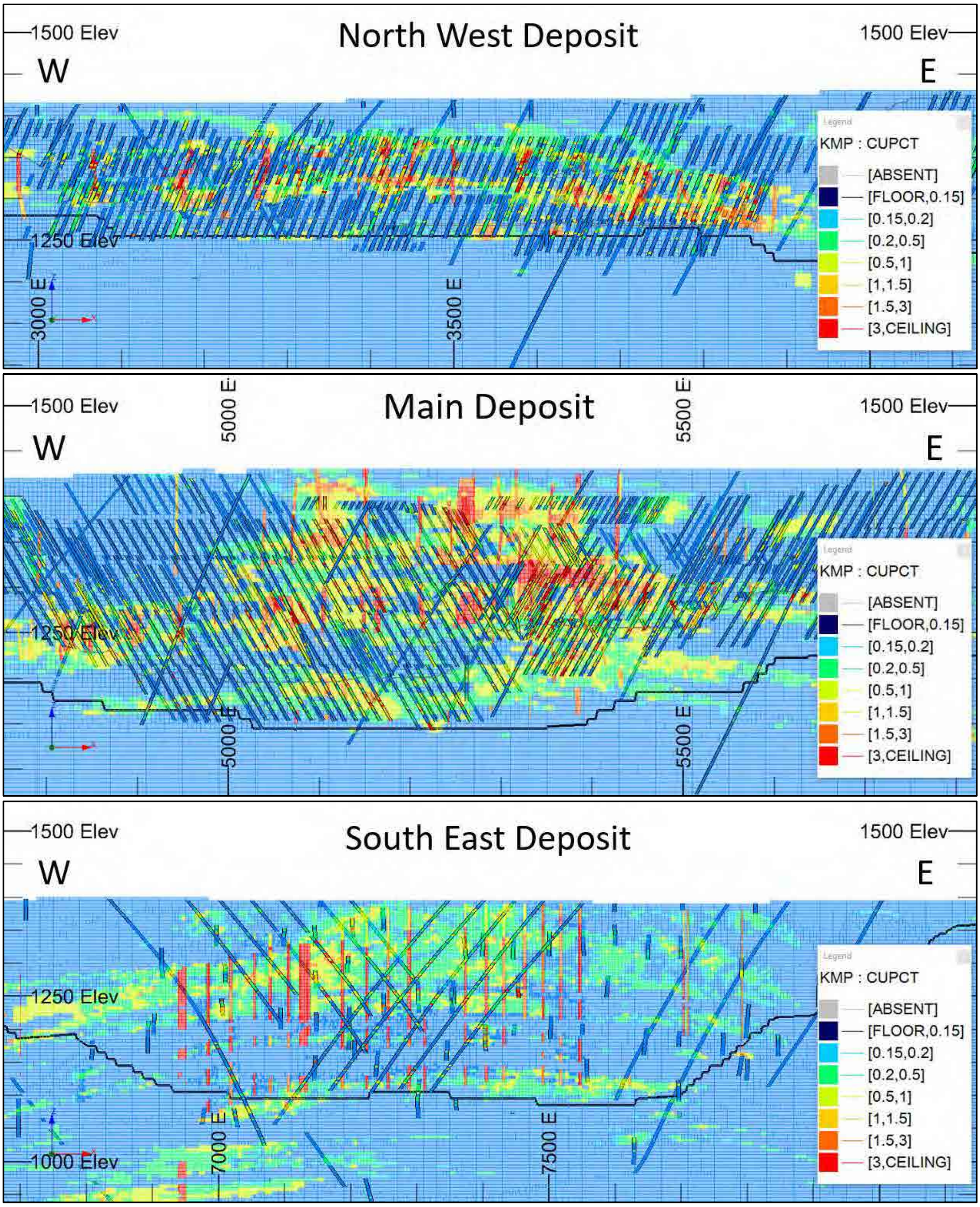
Visual validation indicates that the grade tenor of the input data is well represented in the block model estimates (Figure 14-13).

Vein Cu and Au estimates validate well compared to their input data. Cu estimates were within 4% and Au within 7% of their input sample data. ASCu grades were 20% lower compared to input data means. The lower

ASCu estimates may be attributed to highly variable ASCu grades across the oxidation boundaries, with high grades in oxide and much lower grades in refractory and sulphide.

Stratigraphy grade estimates were generally lower than the input data, attributed to scattered small vein (centimetre scale) grades in areas of lower grades, which smooths these higher grades. Additionally, some lower grade extrapolation around the edges of the deposits with poorer sample support likely influenced these lower grades.

Figure 14-13 Visual validation of the block model estimates and drill data for Cu% values, with vertical sections showing the North West Pit (top), Main Pit (middle) and South East deposit (bottom). Black line indicates the extension of the optimised pit shell (FQM, 2024)



Swath plots (Figure 14-14 to Figure 14-16) proved as an effective method to validate model estimates at a semi-local scale. Swaths were used to validate both veins and major stratigraphy estimates. Overall, swath validations were good, particularly where sufficient data informed the block estimates.

The respective validation steps highlight that the employed estimate methods have adequately represented the input sample data and the prevailing geology.

Figure 14-14 Swath plots for the Main deposit by oxidation domain (mean from drill hole samples in red, estimated mean in black, bars are sample counts

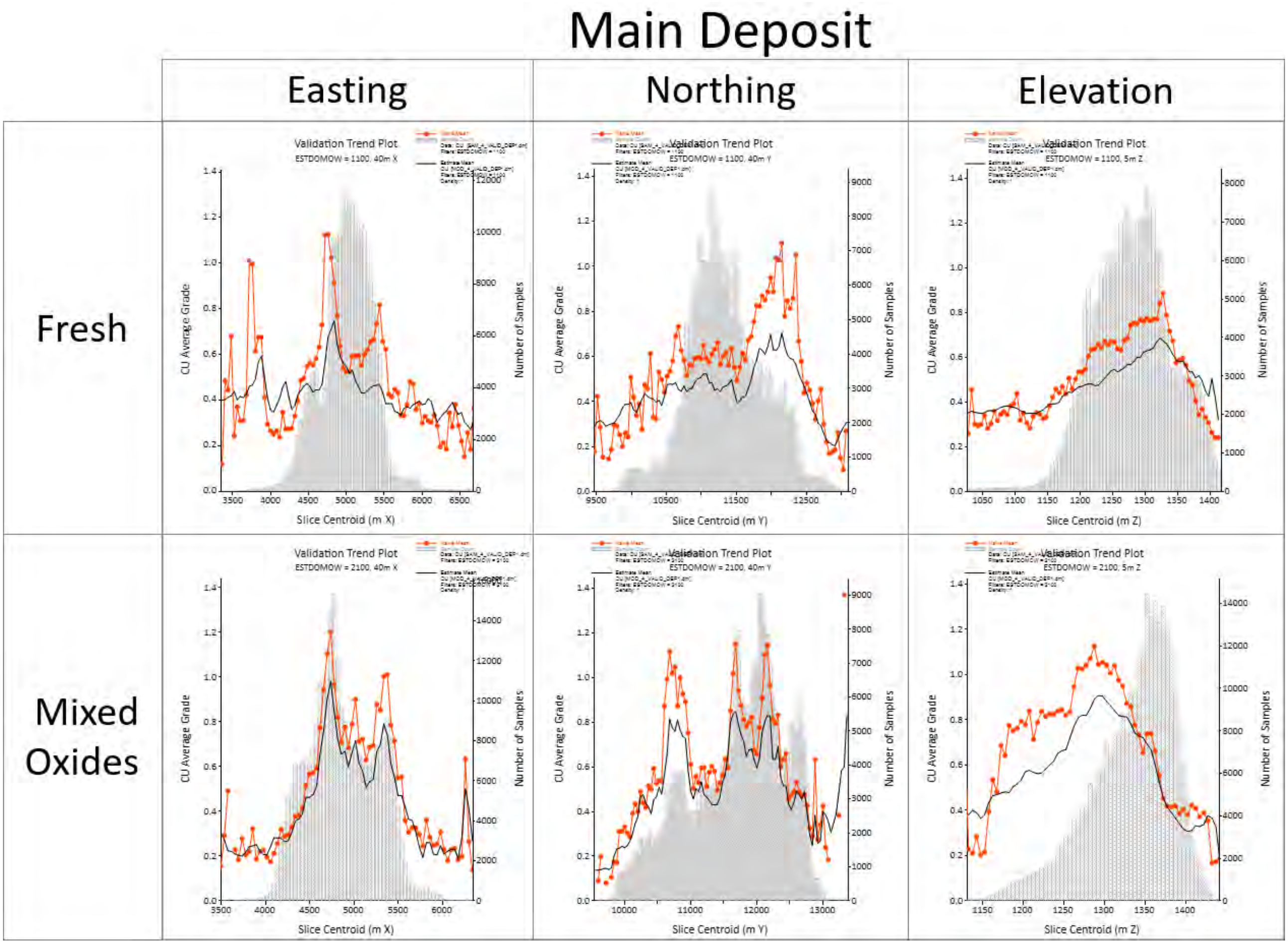


Figure 14-15 Swath plots for the North West deposit by oxidation domain (mean from drill hole samples in red, estimated mean in black, bars are sample counts

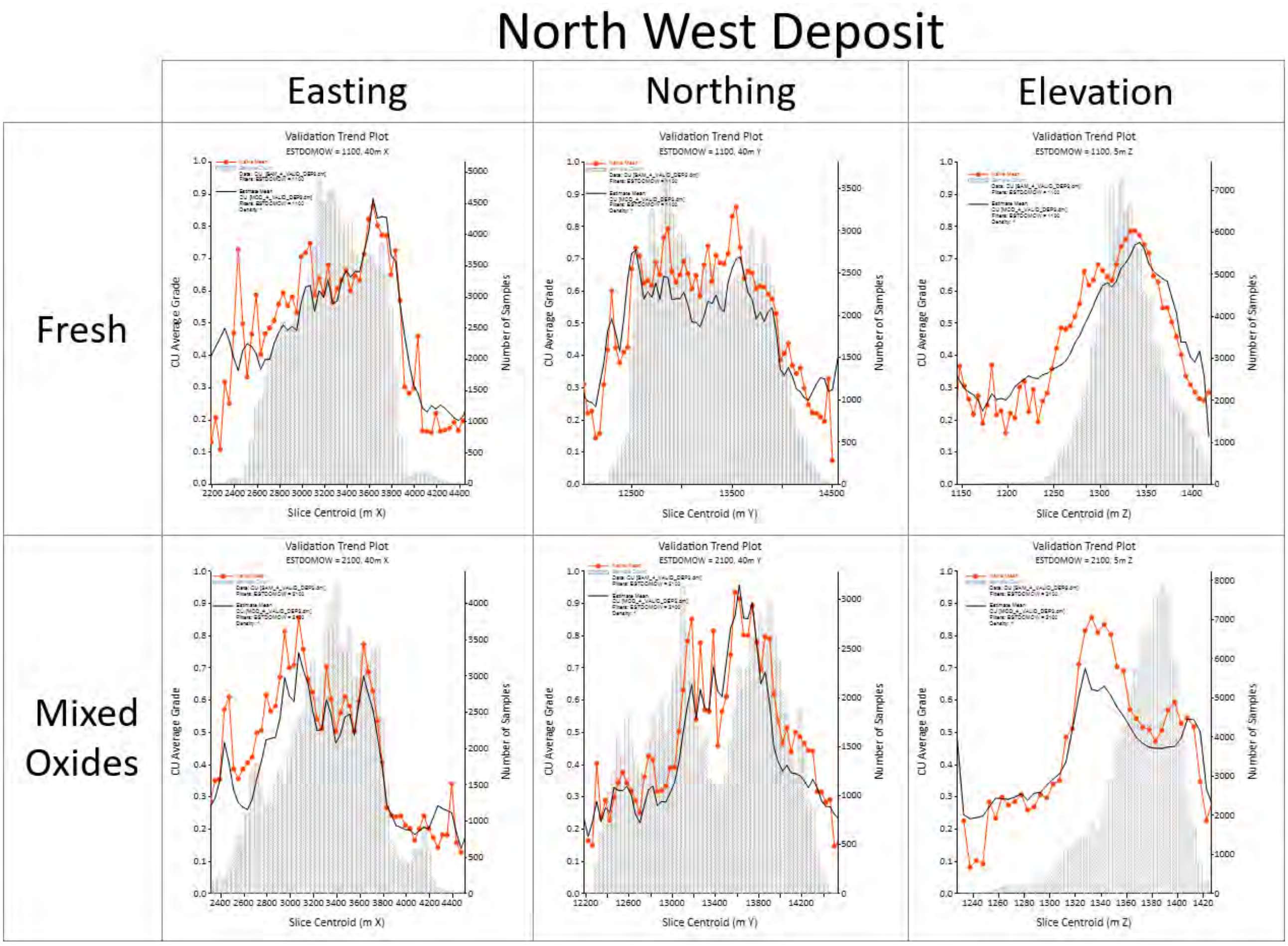
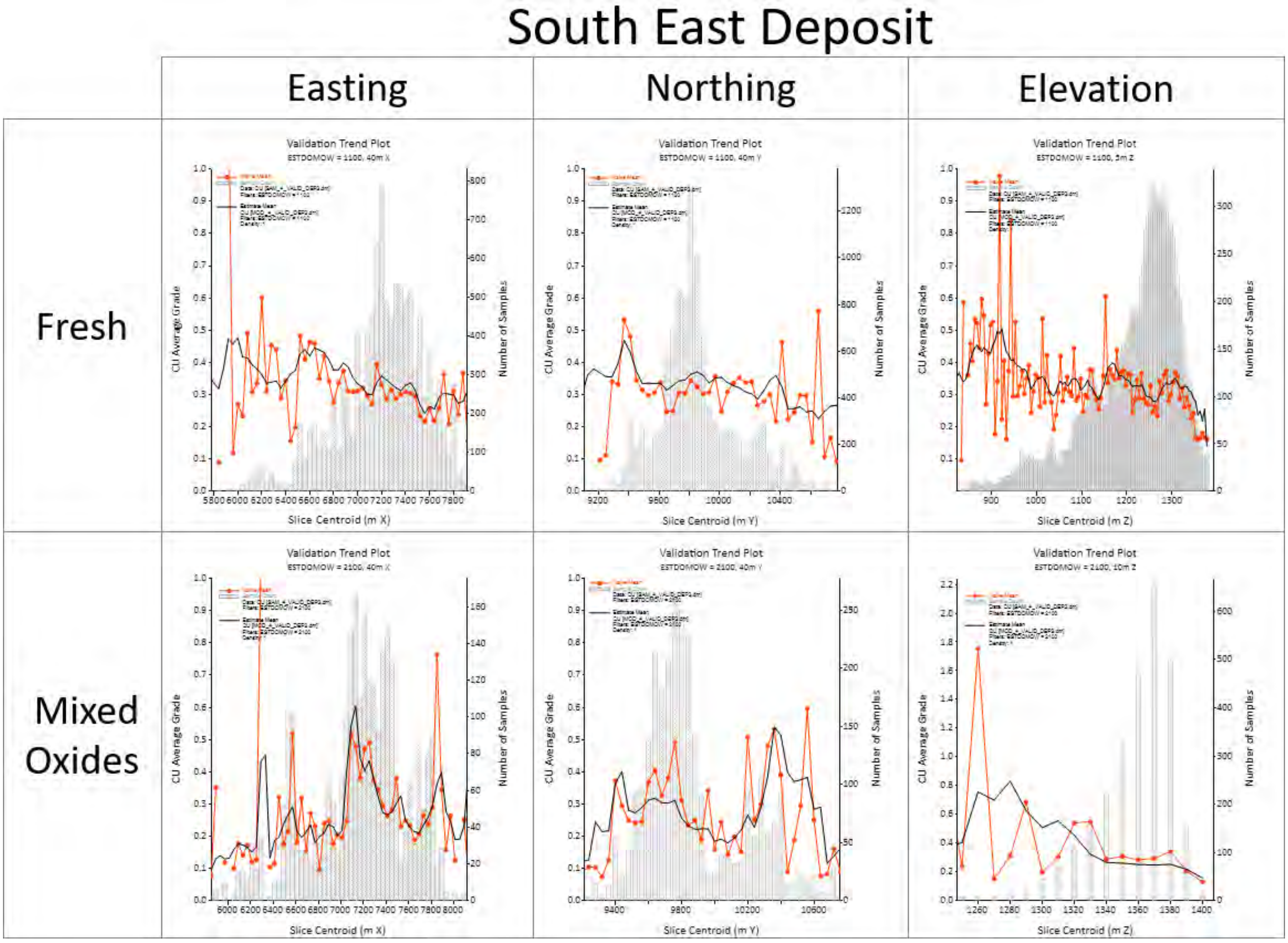


Figure 14-16 Swath plots for the South East deposit by oxidation domain mean from drill hole samples in red, estimated mean in black, bars are sample counts

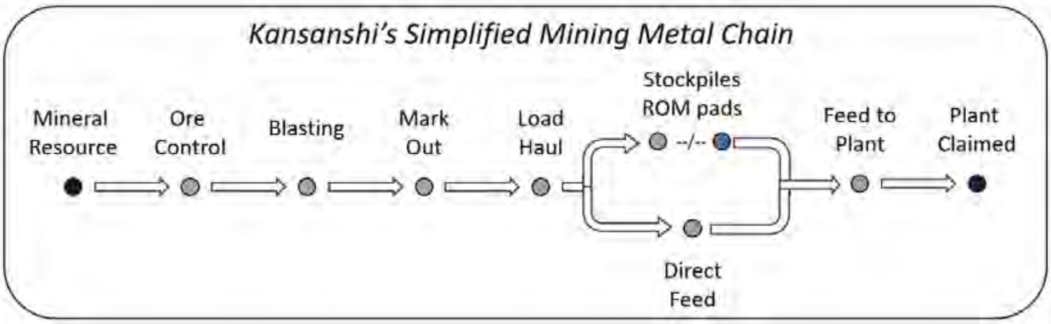


14.17.1 Kansanshi reconciliation

Reconciliation results were used by the Qualified Person as a guide for the Mineral Resource classification, as a validation element, and as support for block estimates to be a reasonable representation of the available metal in the ground. The reconciliation focus was on total copper metal, which is Kansanshi's primary product and economic driver.

Kansanshi's copper production follows a standard mining and processing workflow through the value chain. The in-situ copper metal is impacted by mining processes that mix in-situ ore and waste rock, including material type assignment, blasting, mark-out, loading, haulage, and stockpile feed. Notably, about 40% of the in-situ material mined is fed directly to the plant, while 60% is re-directed to short- and long-term stockpiles. Similarly, the ore feed is comprised of 40% direct feed material and 60% material from stockpiles (Figure 14-17).

Figure 14-17 Kansanshi's simplified Mining Metal Chain



Monthly reconciliation data was compiled for the annual period from March 2023 to the end of February 2024. This period aligns with and captures some of the changes introduced to the Mineral Resource estimates included in this Technical report, as some updates have been developed and implemented into the site processes since the first quarter of 2023. In this comparison:

- The Mineral Resource and Ore Control block model estimates were regularised to the SMU dimension as the input into the Mineral Reserve process, and then evaluated for mined volume based on start and end-of month-survey surfaces.
- Blasting figures evaluate the variance between the input and output models to the blasting process, which uses a muck pile displacement prediction tool.
- Mark-out evaluates the amount of dilution and loss included in the polygons defined for practical mining.
- Material mined from the pit (Ex-pit) uses tonnages from trucks weightometers and grades from mark-out polygons.
- Stockpile figures are based on tonnages from truck weightometers and grades assigned to each stockpile. Stockpile data prior to 2021 does not include blast movement or mining, digging or hauling impacts on tonnages, grades or metal. The stockpile data used in this analysis includes a correction for dilution and losses based on production reconciled data.
- Plant Claimed data uses tonnages and grades at the Mill stage.
- All reported tonnages are dry.

A summary of the reconciliation data is presented in Table 14-34.

Table 14-34 Kansanshi Reconciliation Summary March 2023 - February 2024

Source - Metal Value Chain	Tonnes (Mt)	Cu (%)	Cu Metal (kt)	Metal Var(kt)	% Var.
Mineral Resource Estimate (1)	23.85	0.93	22.30	-0.77	-3.5%
Ore Control Estimate (1)	22.73	0.95	21.52		
Pre-Blast (2)	22.36	0.95	22.76	-1.21	-5.4%
Post-Blast (3)	22.80	0.95	21.55		
Mark-out dilution (3)	23.55	0.87	20.44	-1.11	-5.0%
Feed: Mined Ex Pit + Stockpiles (4)	26.41	0.71	18.81	-1.63	
Feed: Mined Ex Pit impact (40% of feed)				-0.65	-2.9%
Feed: Stockpile Impact (60% of feed)				-0.98	-4.4%
Metal Balance (5)				**-0.35**	**-1.6%**
Plant Claimed	**27.31**	**0.63**	**17.23**	-	
(1) MRE and OC reported at SMU size 10*10*5m (regularized)					
(2) Pre-blast is at the higher resolution of the geological model (unregularised)					
(3) Post blast and Mark-out are reported at 5*5*5m (regularised)					
(4) Feed Metal variance (-1.63 kt) is split 40:60 into Mining Ex-Pit (-0.65kt) and Stockpiles (-0.98kt)					
(5) Metal balance = Marginal error in Ore Control Estimates plus error of measures & equipment					

Key reconciliation results include:

a) The Mineral Resource estimate showed 3.5% more metal than the Ore Control estimate, indicating confidence in the Mineral Resource estimate.

   i) Compared to the Plant claimed, the Mineral Resource estimate showed 22.7% more metal.

b) Blast movement mixing reduced copper metal by 5.4%.

c) Mark-out for practical digging and loading further reduced copper metal by 5.0%.

d) Mining feed to the Plant included approximately 60% of material reclaimed from historical stockpiles, limiting direct reconciliation, with the remaining 40% coming from ex-pit mined material. Stockpile feed will vary as they are depleted.

   i) Selective mining losses during digging, loading, and hauling accounted for approximately 2.9% reduction of copper.

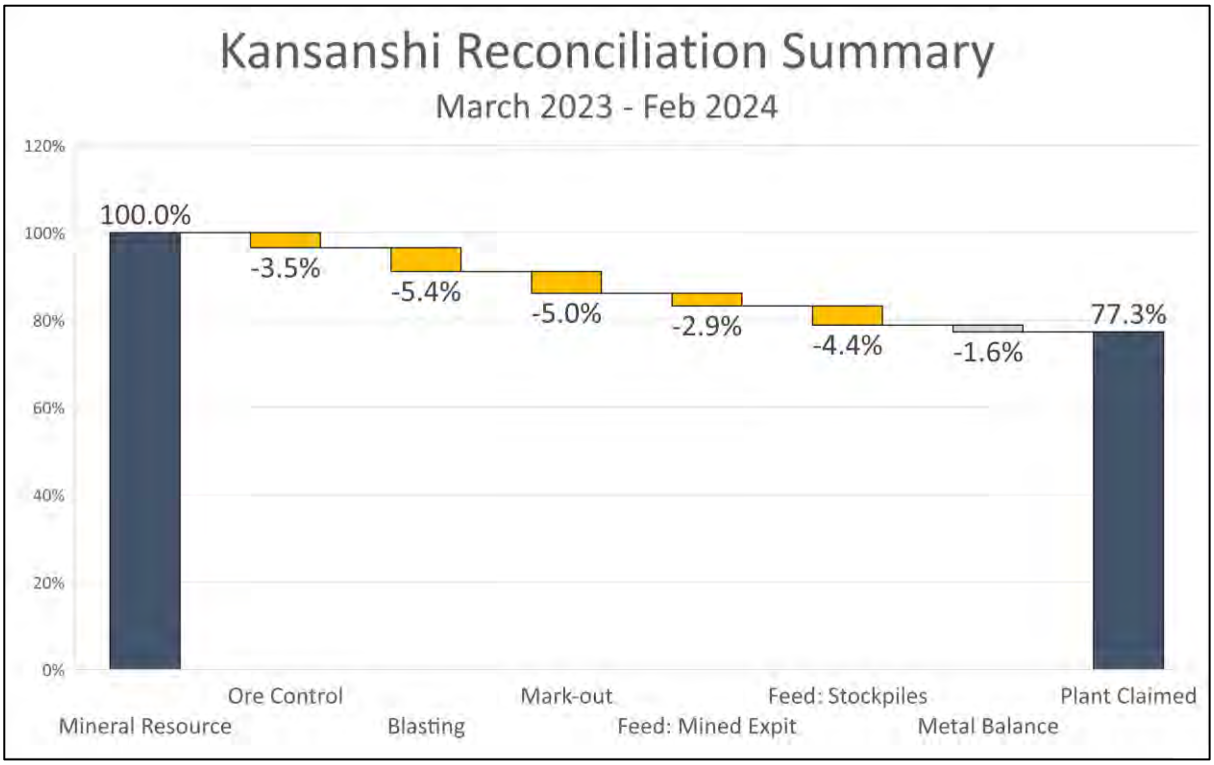
   ii) Stockpile feed, comprising ~60% of the total Plant feed, reduced annual metal by approximately 4.4%. Stockpile tonnes prior to 2021 do not include blast movement metal impacts or the impacts of digging, loading and haulage.

e) The remaining balance against the Plant Claimed metal was likely to be due to a marginal error in the Ore Control estimates of approximately 1.6%. It is likely that this figure will vary according to moisture and weighting variations during mining and stockpile handling.

Each of these variances are believed to reflect reasonable Ore Control practices of ensuring a highest quality tonne as feed to plant.

The reconciliation variances at each step are shown in Figure 14-18 .

Figure 14-18 Kansanshi reconciliation summary waterfall chart for the period March 2023 to February 2024



Mineral Resource estimates are likely overstated by approximately 3.5% compared to Ore Control estimates, which have an increased estimate accuracy. Ore control estimates could also be responsible for the unassigned 1.6% metal balance, i.e., marginal errors in the Ore Control estimates.

In summary, the comparison between Mineral Resource estimates and Plant Claimed figures suggests that confidence in Mineral Resource estimates is well aligned with a Measured Mineral Resource classification performance, with an estimated error of approximately 5.1% for an annual period (Table 14-35).

Table 14-35 Kansanshi Mineral Resource estimate error as obtained from March 2023 to February 2024 reconciliation data analysis

Source	Error (%)
Mineral Resource to Ore Control error	3.50%
Mineral Balance - Marginal OC error	1.60%
Total Mineral Resource Estimate error	**5.10%**

14.18 Resource classification

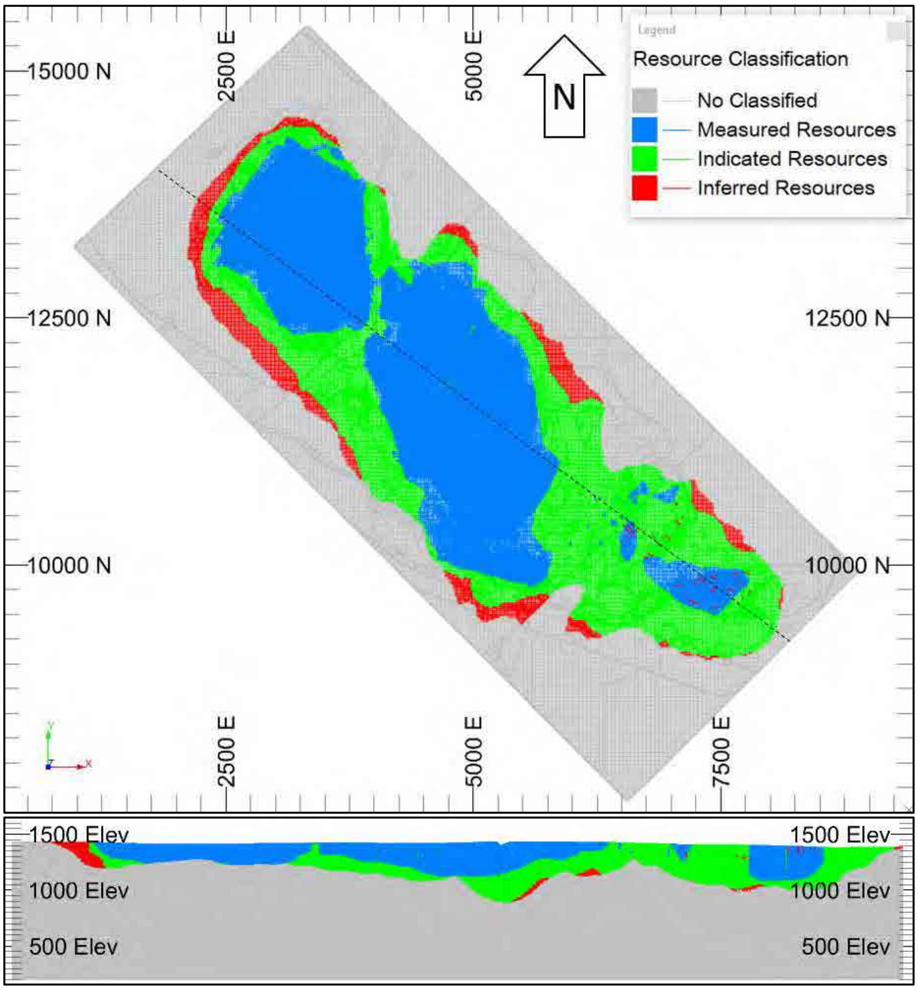
The Mineral Resource estimate was classified as Measured, Indicated and Inferred (Figure 14-19) in accordance with the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines, CIM November 2019), and the guidelines of the JORC Code (JORC, 2012).

Classification was based on several criteria including assessment of the reliability of the geological model, sampling, survey control, bulk density data, drilling grid, kriging statistics, positive metallurgical performance, mine-to-mill reconciliation support, and the reasonable prospect of eventual economic extraction. Specifically, the following criteria were considered during classification by the QP:

- Verification of tenement title, drilling, sampling, and geological process, standards and systems, as completed during site visits by Carmelo Gomez and David Gray between 2020 and 2024.
- Ensuring a reasonable expectation of an economic outcome for classified Mineral Resources by aligning block estimates with Mineral Reserves, mine plans and processing, guided by and controlling mineralised volumes to an ultimate pit shell (based on a $3.90/lb copper price).
- Good geological evidence for continuity of mineralisation at the cut-off grade, as per estimation of the Mineral Resource.
- Excellent open pit exposure and operational performances, as evidence supporting the sample analytical values for copper, acid soluble copper and gold mineralisation.
- Good QAQC controls and results, verifying robust sampling practices and analysis of copper and gold grades (associated to the strata mineralisation).
- Adequate DD core sampling to determine dry in-situ bulk density for applying to estimates for the tonnages of mineralisation.
- Safe, secure databases providing validated data for both DD and RC drilling data.
- Supportive ordinary kriging slope of regression and kriging efficiency values as indicators of relative confidence in the grade estimates.
- Sound mining reconciliation data supporting wide spaced DD estimates as robust when compared with close drilled RC estimates.

Kriging ‘confidence,’ measured by the slope of regression, was combined with geological confidence, sample spacing and the potential for economic extraction, as a guide to determining classification boundaries.

Figure 14-19 Plan (top) and cross section view (bottom) of the Kansanshi block model, coloured by Mineral Resource classification (FQM, 2024)



14.18.1 Classification of strata Mineral Resources

- Measured Mineral Resource wireframes were constructed to delineate blocks estimated in the first search pass with a regression slope value above 0.8. This correlates with areas where drill grid spacing and data support is within a 50 m x 50 m x 30 m grid.
  - Mineralised volumes within closed spaced areas drilled for Ore Control (RC drilling at 12m x 12.5m, or smaller) were classified as Measured.
- Indicated Mineral Resource wireframes were constructed to delineate blocks estimated in the first or second search pass with a regression slope value above 0.6. These volumes correlated with a drilling grid spacing of 100 m x 100 m x 30 m.
- Inferred Mineral Resource wireframes were constructed to flag the remaining block estimates that did not meet the Measured or Indicated criteria but where geological and grade continuity persisted with reasonable confidence. These estimates were close to 300 m below surface and the current ultimate pit shell using a potential $3.90/lb copper metal price. Mineralised diorite was also classified as Inferred.

14.18.2 Classification of vein Mineral Resources

VNAREA = 0

- Vein Mineral Resources in VNAREA = 0 were classified the same as their respective strata parent blocks.
- This vein mineralisation will be mined within the context of each parent strata mineralisation.

VNAREA = 1

Vein Mineral Resources were classified according to their estimate pass and parameters, from higher to lower confidence levels.

- Measured classification was applied to vein blocks estimated in the first search pass using more than 14 samples for each estimate. The first past search is on average 130 m (along vein strike) by 95 m (along vein dip plane) and 8 m across.
- Indicated classification was applied to vein blocks that estimated in the first search pass with 8 to 14 samples and to blocks estimated within the second search pass.
- No Inferred classification was applied to vein blocks within the VNAREA = 1, which aligns with the area covered by close spaced reverse circulation drilling (at 12.5 m x 15m grid).

VNAREA = 2

- No Measured classification was applied to vein Mineral Resources in South East deposit.
- Indicated classification was applied to vein blocks estimated in the first or second search pass. The first pass was 150 m by 50 m by 10 m using a minimum of 8 samples.
- Inferred classification was applied to blocks not estimated during process, which received the average grades per domain.

14.19 Mineral Resource reporting

The updated Mineral Resource estimate for Kansanshi is presented in Table 14-36. The Mineral Resource estimate block model has been regularised to an SMU of 10m x 10m x 5m and is reported at a 0.2% total copper cut-off grade, consistent with the applicable cut-off grade for the Mineral Reserve estimate. The Mineral Resource estimate has been depleted of mined material as at 31st December 2023. Weathered laterite and refractory material were not included due to processing risks and the subjective quality of available logging data. Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 14-36 Kansanshi Mineral Resource statement, excluding stockpiles, at a 0.2% Total Copper cut-off and depleted of mined material as at 31st December 2023

Classification	Volume (millions)	Density (t/m^3)	Tonnes (millions)	Cu (1) (%)	ASCu (%)	Au (g/t)	Cu metal (1) (kt)	ASCu metal (kt)	Au metal (2) (koz)
Measured	164.1	2.57	422.0	0.68	0.12	0.12	2,881.0	520.3	1,674.2
Indicated	270.6	2.73	738.9	0.57	0.06	0.12	4,244.5	429.0	2,811.2
Total Measured & Indicated	**434.7**	**2.67**	**1,160.9**	**0.61**	**0.08**	**0.12**	**7,125.5**	**949.4**	**4,485.5**

Classification	Volume (millions)	Density (t/m³)	Tonnes (millions)	Cu (1) (%)	ASCu (%)	Au (g/t)	Cu metal (1) (kt)	ASCu metal (kt)	Au metal (2) (koz)
Inferred	17.9	2.75	49.3	0.41	0.02	0.09	200.2	11.1	143.9
(1) Cu (%) grade is inclusive of ASCu (%) grade. Cu metal (kt) is inclusive of ASCu metal (kt).									
(2) 1 troy ounce= 31.1035 grams									
Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability									

Notes:

- The Mineral Resource estimate was prepared by Mr. Carmelo Gomez, B. Sc. (Hons), EurGeol, an FQM (Australia) Pty Ltd employee, and the Qualified Person for the estimate.
- Mineral Resources have an effective date of 31st December 2023.
- Material depleted as of 31st December 2023 has been excluded from this Mineral Resource estimate statement.
- Block model grade interpolation was undertaken using ordinary kriging (OK) for all metals with the exception of Au in the strata mineralisation domain, which used multi indicator kriging (MIK).
- Dry bulk density was estimated by strata and weathering domain using ordinary kriging.
- The Mineral Resources were estimated using the “CIM Definition Standards for Mineral Resources and Mineral Reserves” of 10 May 2014 and the “CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines” of 29 Nov 2019, as prepared by the CIM Standing Committee and adopted by CIM Council.
- Resource classification is as defined by the Canadian Institute of Mining, Metallurgy and Petroleum in their document “CIM Definition Standards for Mineral Resources and Mineral Reserves” of 10 May 2014.
- Mineral Resources are presented on a 100% basis.
- Mineral Resources are inclusive of Mineral Reserves.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- Tonnage and grade figures have been rounded to reflect the relative accuracy of the Mineral Resource estimate as required by reporting guidelines; therefore, columns may not total due to rounding.
- Open pit cut-off grade used for Mineral Resource estimate reporting is 0.20% Cu.

To the best knowledge of the QP, Carmelo Gomez, 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 having reasonable prospects for economic extraction.

Grade and tonnage data for the Measured and Indicated Mineral Resources are presented as graphs and tabulated values in Figure 14-20 and Table 14-37.

Figure 14-20 Grade tonnage curve data for Measured and Indicated Mineral Resources depleted of mined material as at 31st December 2023

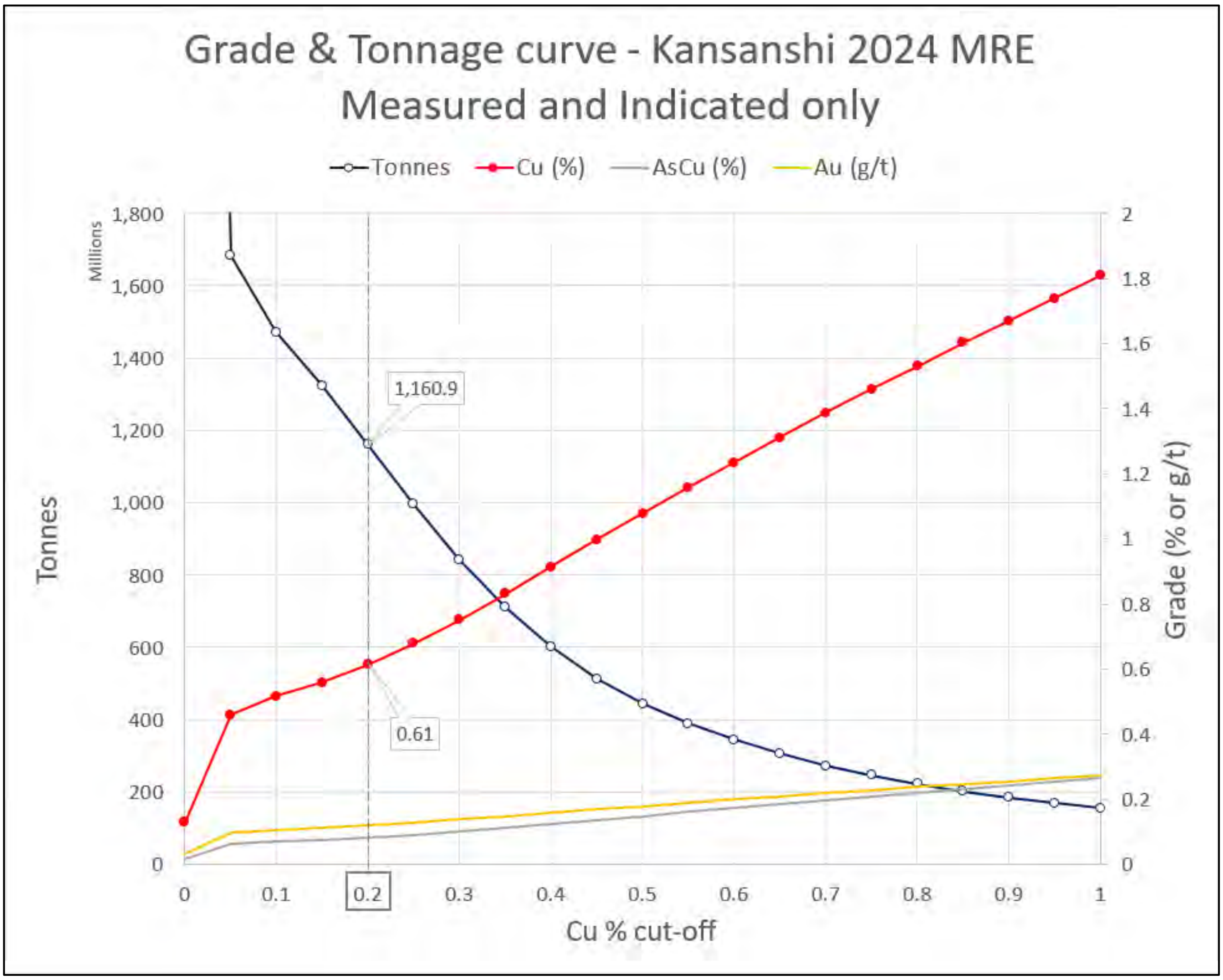


Table 14-37 Grade tonnage curve data for Measured and Indicated Mineral Resources depleted of mined material as at 31st December 2023

Cu % cut-off	Volume (Mm^3)	Density (t/m^3)	Tonnes (Mt)	Cu (%)	ASCu (%)	Au (g/t)
0.00	2,474.8	2.70	6,683.7	0.13	0.02	0.03
0.05	641.7	2.62	1,682.5	0.46	0.06	0.10
0.10	557.9	2.64	1,471.5	0.52	0.07	0.11
0.15	498.9	2.65	1,324.4	0.56	0.08	0.11
0.20	**434.7**	**2.67**	**1,160.9**	**0.61**	**0.08**	**0.12**
0.25	371.4	2.68	995.6	0.68	0.09	0.13
0.30	313.2	2.69	841.5	0.75	0.10	0.14
0.35	264.1	2.69	710.4	0.83	0.11	0.15
0.40	223.4	2.69	601.6	0.91	0.12	0.16
0.45	190.4	2.70	513.3	1.00	0.14	0.17
0.50	165.0	2.70	445.3	1.08	0.15	0.18
0.55	144.4	2.70	389.9	1.16	0.16	0.19
0.60	127.3	2.70	343.9	1.24	0.17	0.20
0.65	113.3	2.70	306.4	1.31	0.18	0.21
0.70	101.0	2.71	273.2	1.39	0.20	0.22
0.75	91.1	2.71	246.5	1.46	0.21	0.23
0.80	82.5	2.71	223.3	1.53	0.22	0.24
0.85	74.8	2.71	202.5	1.60	0.23	0.25
0.90	68.4	2.71	185.4	1.67	0.24	0.26
0.95	62.7	2.71	169.9	1.74	0.25	0.26
1.00	57.4	2.71	155.6	1.81	0.26	0.27

In addition to the unmined Mineral Resources, several stockpiles have been generated during the open pit mining operation. The current stockpiles include separate areas for oxide, mixed and sulphide materials and are classified as Indicated Mineral Resources (Table 14-38).

Table 14-38 Kansanshi Mineral Resource statement for stockpiles as at 31st December 2023

Stockpiles	Classification	Tonnes (Mt)	Cu [(1)] (%)	ASCu (%)
Oxide (float/leach feed)	Indicated	54.1	0.30	0.11
Mixed (float/leach feed)		45.7	0.57	0.18
Sulphide (float feed)		69.7	0.36	0.01
Total Stockpile - Indicated		169.5	0.40	0.09
[(1)] Cu (%) grade is inclusive of ASCu (%) grade. Cu metal (kt) is inclusive of ASCu metal (kt).				
Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.				

The material was mined from areas with closely spaced grade control drilling. The grades of the stockpiles were determined from the open pit grade control model, which utilised the same samples and grades as the Mineral Resource estimate. The marked-out polygons for mining accounted for dilution and loss of metals with adjacent material types. The position of these materials within the stockpiles is not recorded. An ore control database contains detailed information for each polygon, including the date, blast identity, material type, tonnes, moisture and grades, which support the assignment of stockpile average grades.

14.20 Comparison with previous Mineral Resource estimate

The 2024 Mineral Resource estimate incorporates additional diamond drilling data, grade control drilling (RC drill holes), and improved geological models derived from drill core and reverse circulation chip logging, as well as insights gained from pit mapping and metal chain reconciliation from mining operations and material processing conducted over the past years.

A comparison with the 2020 Mineral Resource estimate is detailed in Table 14-39 to Table 14-42, utilising the same reporting cut-off copper grade (0.2 Cu %). In summary:

- The combined tonnes of Measured and Indicated Mineral Resources have increased by 43%, with a 6% decrease in copper grade. The increase in tonnage primarily stems from the addition of DD and RC drill hole data, leading the upgrade of previous Inferred and Indicated Mineral Resources.
- Inferred Mineral Resource tonnes have decreased by 70% as a result of the upgrading of higher-grade material to Indicated and Measured categories. The remaining Inferred material has decreased by 30% in copper grade, with the potential for some to be upgraded and included in future Mineral Reserve estimates.

Vein modelling, particularly in the Main and South East deposits, has been enhanced through the addition of significant new pit mapping data, improved drill hole logging information, assays and geological interpretations. These improvements in drilling and geological data have resulted in upgrades to Mineral Resources classification, leading to an overall increase in Measured and Indicated categories.

Table 14-39 February 2024 Mineral Resource estimate depleted as of end of Dec 2023 and using a 0.2 Cu% reporting cut off

Classification	Volume (millions)	Density (t/m^3)	Tonnes (millions)	Cu (1) (%)	AsCu (%)	Au (g/t)
Measured	164.1	2.57	422.0	0.68	0.12	0.12
Indicated	270.6	2.73	738.9	0.57	0.06	0.12
Total Measured & Indicated	**434.7**	**2.67**	**1,160.9**	**0.61**	**0.08**	**0.12**
Inferred	17.9	2.75	49.3	0.41	0.02	0.09
(1) Cu (%) grade is inclusive of AsCu (%) grade.						
Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.						

Table 14-40 September 2020 Mineral Resource estimate depleted as of end of Dec 2023 and using a 0.2 Cu% reporting cut off

Classification	Volume (millions)	Density (t/m3)	Tonnes (millions)	Cu (1) (%)	AsCu (%)	Au (g/t)
Measured	82.0	2.50	204.8	0.66	0.12	0.11
Indicated	223.3	2.73	609.1	0.65	0.07	0.12
Total Measured & Indicated	**305.4**	**2.67**	**814.0**	**0.65**	**0.08**	**0.12**
Inferred	60.4	2.75	166.4	0.58	0.04	0.11
(1) Cu (%) grade is inclusive of AsCu (%) grade.						
Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.						

Table 14-41 Variance of 2024 estimate to 2020 estimate as of end of Dec 2023

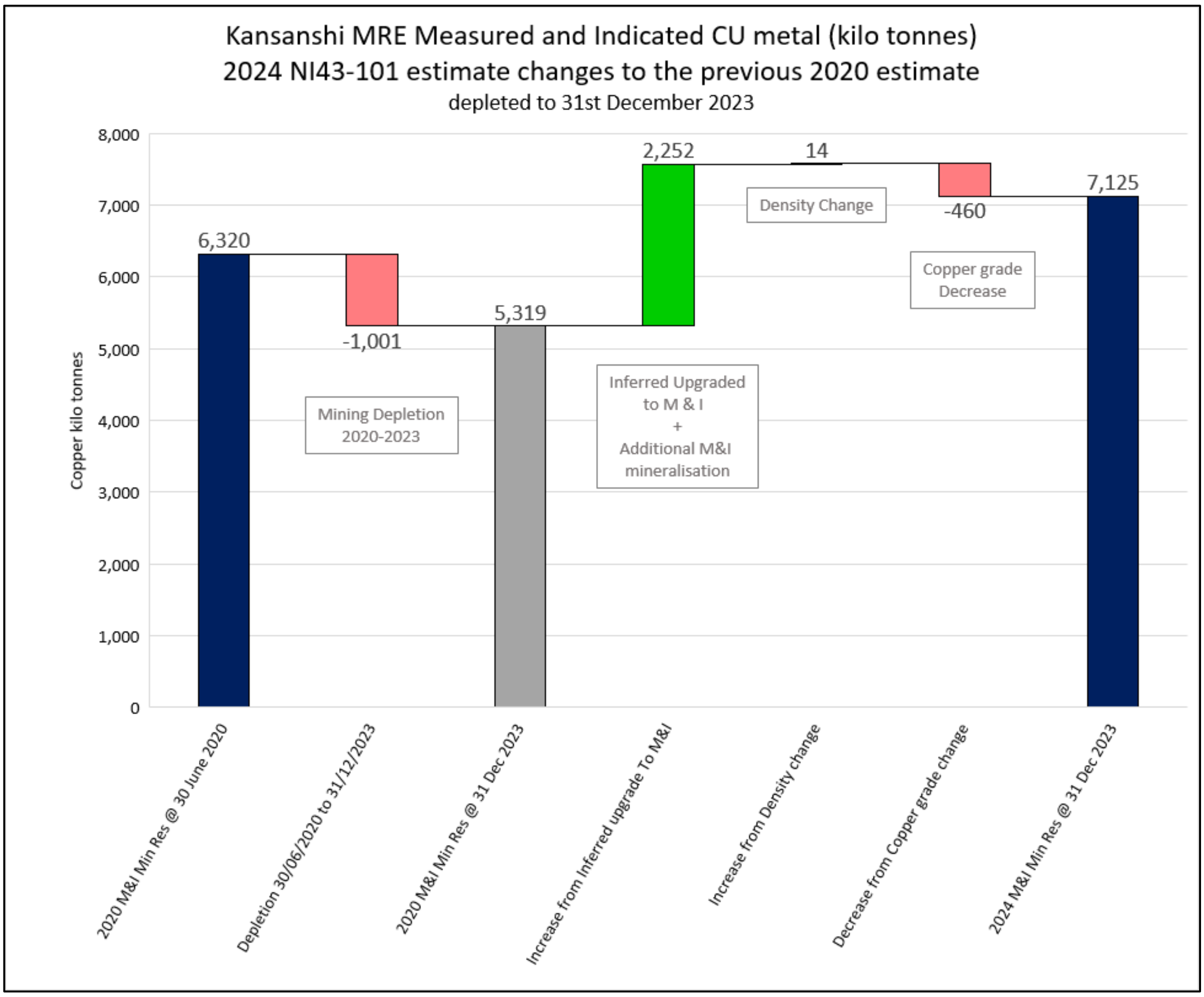
Classification	Volume (millions)	Density (t/m^3)	Tonnes (millions)	Cu (1) (%)	AsCu (%)	Au (g/t)
Measured	82.1	0.07	217.1	0.02	0.00	0.01
Indicated	47.2	0.00	129.8	-0.08	-0.01	0.00
Total Measured & Indicated	**129.3**	**0.00**	**346.9**	**-0.04**	**0.00**	**0.00**
Inferred	-42.5	0.00	-117.1	-0.17	-0.02	-0.02

Table 14-42 Percentage variance of 2024 estimate to 2020 estimate as of end of Dec 2023

Classification	Volume Diff. (%)	Density Diff. (%)	Tonnes Diff. (%)	Cu Diff. (%)	AsCu Diff. (%)	Au Diff. (%)
Measured	100%	3%	106%	3%	3%	10%
Indicated	21%	0%	21%	-12%	-16%	-3%
Total Measured & Indicated	**42%**	**0%**	**43%**	**-6%**	**0%**	**1%**
Inferred	-70%	0%	-70%	-30%	-47%	-21%

Key changes resulting from this Mineral Resource estimate update since the 2020 Technical Report are summarised in a waterfall chart (Figure 14-21). Between the end of June 2020 and the end of December 2023, mining depletions and the positive impact of increased confidence to the estimates from additional geological information, pit mapping and drilling data (302 DD holes and 54,648 RC holes), have been significant contributors to the changes observed in the 2024 Mineral Resources.

Figure 14-21 A waterfall chart of Measured and Indicated copper metal (kilo tonnes), illustrating key changes to Mineral Resource estimates since the 2020 estimate



The QP, Carmelo Gomez, considers this Mineral Resource estimate to be robust and representative of its input data and the prevailing geology. The detail grade and geology knowledge acquired from mining activities, along with the additional DD and RC drilling data, supports this estimate and the resulting Mineral Resource classification. Model comparisons are deemed justified and considered to be acceptable within the context of the confidence associated with the previous Mineral Resource classification.

ITEM 15 MINERAL RESERVE ESTIMATES

Detailed technical information provided under this item relates specifically to the Mineral Reserve estimate completed to date and derived from the Mineral Resource model and estimate as reported in Item 14.

As part of the estimation process, the mine planning work completed for the 2020 Technical Report was reviewed and updated. Along with recent detailed pit designs completed by FQM personnel, subsequent production scheduling work (Item 16) was overseen by Michael Lawlor (QP) of FQM.

To conform to NI 43-101 standards, the Mineral Reserve estimate is derived from Measured and Indicated Resources only. The Measured and Indicated Mineral Resource estimate as listed in Table 14-36 is reported inclusive of the Mineral Reserve.

15.1 Introduction

Mine planning and Mineral Reserve estimation and reporting in the 2015 Technical Report addressed an increase in mining material movements arising from the proposed S3 expansion and the requirement for supplementary plant feed from the South East Dome Pit, commencing from 2017. However, the S3 expansion and development plans for South East Dome were both deferred in late 2015.

Subsequently, in the 2020 Technical Report, the reported Mineral Reserve inventory was increased by approximately 70%, essentially attributable to an updated Mineral Resource estimate and a partial resource reclassification from Inferred to Indicated status. At June 2020, it was envisaged that the S3 construction Project would be resumed, with production commencing in H2 2024 and continuing through to 2044. Pre-strip mining at South East Dome was proposed for 2024, with feed contribution to the S3 plant continuing through to 2040.

Despite these timing projections, formal Company board approval for restarting the S3 Project was not forthcoming until May 2022. In the meantime, continuing Mineral Resource estimations have benefited from modelling and mineralisation improvements which translate to further enhancements to the Mineral Reserve inventory.

First ore to the S3 crushers is now scheduled for Q2 2025. Ore mining from South East Dome will commence in 2024.

15.2 Methodology

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

A selected pit optimisation shell was used to guide the creation of practical and detailed open pit stage (i.e., cutback) designs accounting for pit batters, berms and haul roads.

These pit cutback 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 (Section 16.14).

15.3 Mine planning model

A mine planning model was produced from the Datamine Mineral Resource model described in Item 14.

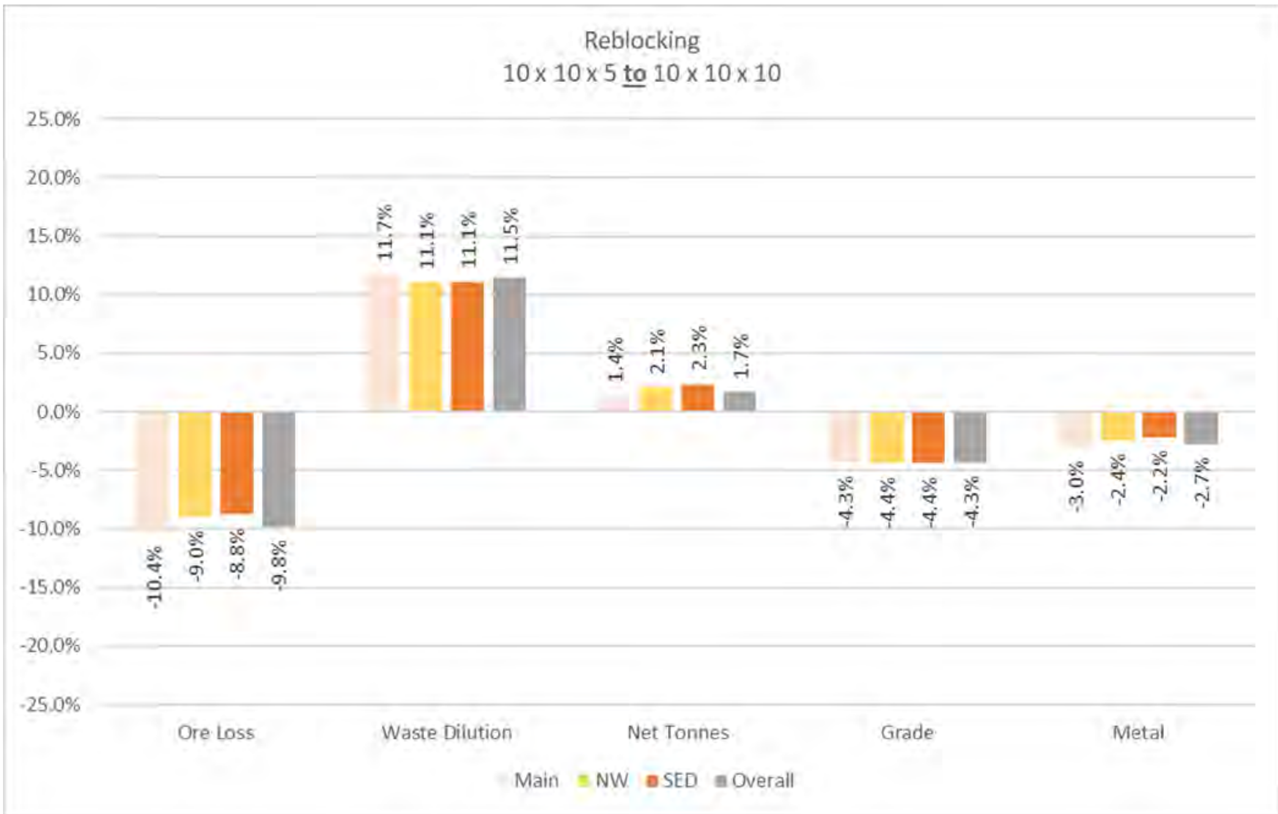
Similar to the approach taken for the 2020 Technical Report, and as a first step, the resource model was reblocked to account for the mining SMU (selective mining unit) size. In 2020, the reblocking was to a 10 m x 10 m x 5 m resolution, whilst in this instance the resolution was to 10 m x 10 m x 5 m (for current NW and Main Pit cutbacks) which will be mined in 5 m flitches, and 10 m x 10 m x 10 m to account for the larger mining equipment to be deployed into future full bench height mine cutbacks. In essence, original blocks of 5m height are merged into a single 10 m height block in the 10 m x 10 m x 10 m model.

15.3.1 Planned mining dilution and losses

Mining dilution is incurred where merged ore and waste blocks have an average grade above the cut-off grade, meaning the waste block is thereby incorporated. Mining losses occur when the combined average grade of a merged block is lower than the cut-off grade and the block is thereby defined as waste.

The impact of reblocking from the 10 m x 10 m x 5 m Mineral Resource model to a 10 m x 10 m x 10 m size is shown in Figure 15-1. The chart indicates a 1.4% to 2.3% increase in tonnage dilution from deposit to deposit, with commensurate losses in insitu copper metal of between 2.2% and 3.0%[6].

Figure 15-1 Impact of model reblocking to 10m x 10m x 10m resolution



In the production scheduling process supporting the Mineral Reserve estimate (Item 16.14), the reblocked ore SMU parcels may not always present as contiguous shapes that can be practically mined. In a second step, which essentially emulates the orderly mark-out of mining blocks on a mining bench surface, a routine within the mine planning process "smooths" and groups the SMU blocks into practical mineable parcels. In this step, further mining dilution is added from the expansion of vein mineralisation (i.e. to practical dimensions for mining) and through the incorporation of any isolated waste blocks within an ore mining parcel. Losses can be incurred when isolated ore blocks are located within a waste mining parcel. The impact of the smoothing process is shown in Figure 15-2. The smoothing process adds an additional 4.5% to 7.7% in tonnage dilution, a 4.5% - 7.2% reduction in grade with a further net loss in copper metal of 0.3%.

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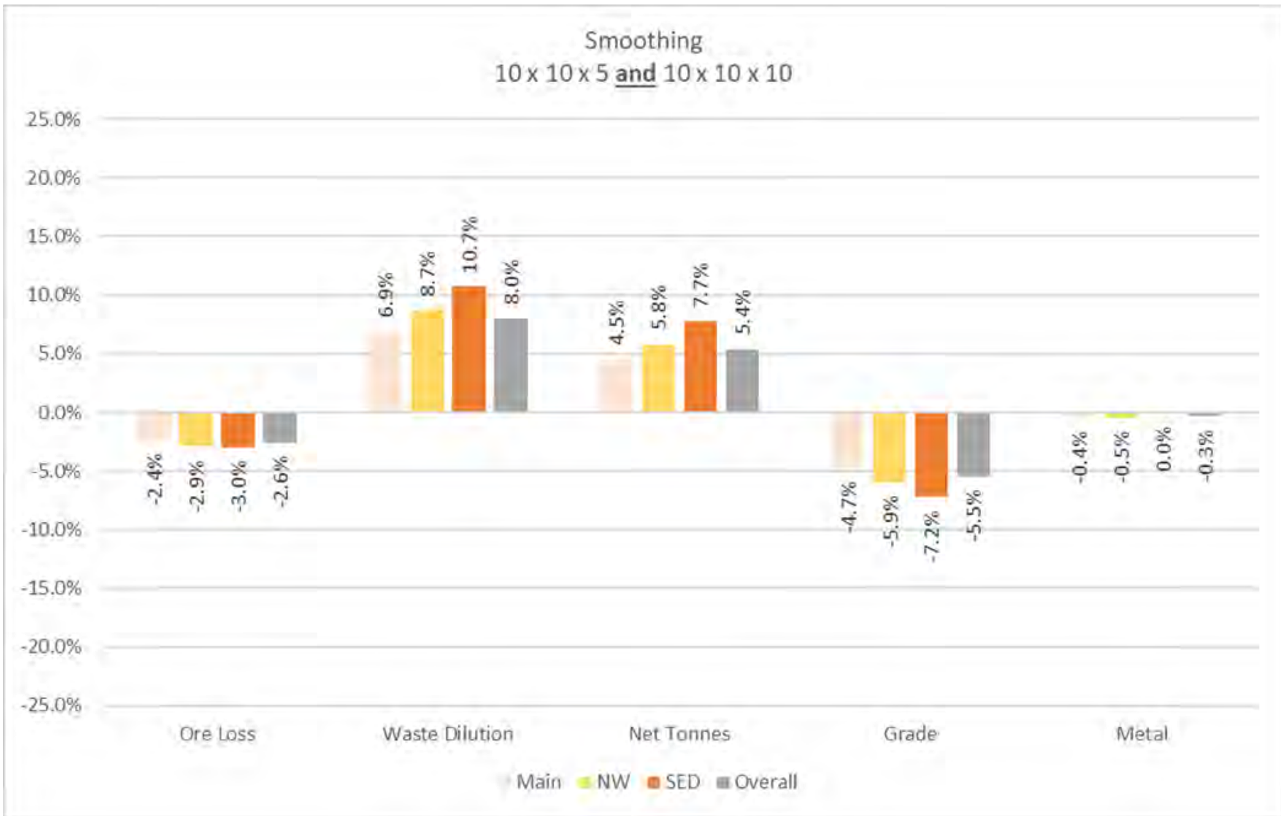
[6] The combined reblocking and smoothing steps can be applied to all future cutbacks where the larger mining equipment will operate. Certain cutbacks that are currently being mined will continue to use smaller equipment for flitch mining. Hence, model reblocking to the 10 m height dimension was not adopted for these particular cutbacks.

Figure 15-2 Impact of model smoothing



In this two-step reblocking and smoothing process, the total mining dilution and losses incurred in producing the mine planning model are referred to as "planned" dilution and losses. Some of the ore lost in the reblocking process can be added back during the smoothing step, and similarly for mining dilution. The overall average (i.e., combined Main, North West and South East Dome Pits) impact of reblocking and smoothing is listed in Table 15-1.

Table 15-1 Overall average, weighted planned mining dilution and loss impact

Resource Model Resolution	10 m x 10 m x 5 m	
Planning Model Resolution	10 m x 10 m x 10 m	
	Tonnes	Cu metal
Reblocking process		
modelled dilution (average)	11.5%	1.4%
modelled losses (average)	-9.8%	-4.1%
combined impact	1.7%	-2.7%
Smoothing process		
modelled dilution (average)	8.0%	1.0%
modelled losses (average)	-2.6%	-1.4%
combined impact	5.4%	-0.3%
Total "planned" factors (overall average)		
modelled dilution (average)	**19.5%**	**2.4%**
modelled losses (average)	**-12.4%**	**-5.5%**
combined impact	**7.2%**	**-3.1%**

15.3.2 Unplanned mining dilution and losses

A third step in the mine planning model process includes an allowance for "unplanned" mining dilution and losses, i.e. those occurring after the bench mark-out and with blasted ore presented for loading and hauling. These dilution and loss allowances account for incorrect loading and hauling practice, i.e. over/under digging and hauling to an incorrect destination. The blasted muck pile, in itself, is also another source of unplanned dilution and mining recovery losses, due to the redistribution and intermixing of ore and waste due to blast heave and throw.

Varying unplanned allowances were applied over the life of mine, including those incorporated through the adoption of the currently budgeted five-year plan. Allowances thereafter were tempered with indications from production tracking records for 2023 and with an expectation of operational improvements over the remaining life of mine timeframe. These allowances were derived through a mathematical weighting process to yield unplanned metal losses (i.e. reflective of combined tonnage dilution, tonnage losses, diluent copper grade and hence overall copper metal loss) varying over time between a high of 12.4% and a low of 1.0%.

Unplanned losses have also been assigned to stockpile reclaim tonnages. Strictly speaking, this allowance is to account for stockpile grade uncertainty, rather than dilution due to rehandling from a stockpile. [The mined tonnage going onto a stockpile will have already had unplanned losses assigned]. When stockpile reclaim is accounted for in the mathematical weighting process, the unplanned metal losses over time range between a high of 9.2% and a low of 0.7%.

A discussion on the methodology adopted for assigning "unplanned" metal loss allowances is provided in Item 16.4.2.

15.3.3 Alignment adjustment

An additional loss adjustment has been incorporated into the mine planning model in order to properly align and merge the contained copper metal profile with that in the detailed internal budget plan for the period 2024 to 2028. The additional unplanned loss adjustments range from 14.7% in 2024 to 2.0% in 2029.

Further discussion and charts depicting the application of this adjustment and the transition to the longer term unplanned loss profile is provided in Item 16.4.2.

15.3.4 Metal loss risk

There is also a consideration of metal loss risk incorporated into the process of developing the mine planning model, relative to the proportions of vein, stratiform and composite mineralisation styles that are apparent in the original Mineral Resource model. An applicable risk matrix is shown in Figure 15-3 and can be construed as the risk of metal loss (through mining dilution and mining recovery losses) when mining different domains, each with varying volumes of mineralisation.

Figure 15-3 Risk matrix, insitu metal volumes within different mineralisation styles

MINING METAL RISK		Strata / Vein metal domains				
		Strata only	Strata dominant	Strata and Vein	Vein dominant	Vein only
MINERALISATION VOLUMEN %	<25%	MEDIUM	MEDIUM	HIGH	HIGH	HIGH
	25%-50%	LOW	MEDIUM	MEDIUM	HIGH	HIGH
	50%-75%	LOW	LOW	MEDIUM	MEDIUM	HIGH
	>75%	LOW	LOW	LOW	MEDIUM	MEDIUM

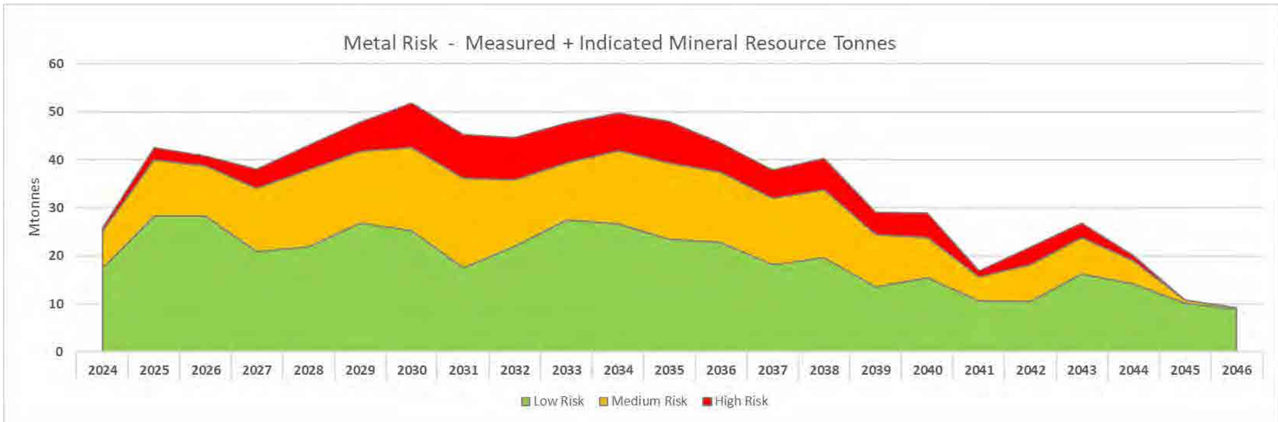
This risk dimension can be shown across the LOM timeframe, in the context of the Mineral Resource tonnage classifications (and by inference, to the Mineral Reserve inventory), and according to the associated plant feed types. Figure 15-4 indicates that, in terms of potential mining metal losses:

- for the combined resource classification, over the LOM period, an average of 58% of the resource tonnage would be in the low risk category

- for the combined classification, over the LOM period, an average of 29% of the resource tonnage would be in the medium risk category
- hence, 13% of the resource tonnage over the LOM period would be in the high risk category
- for the combined classification, the peak high risk years are between 2029 and 2033, representing an average of 18% of the resource tonnage
- during this peak high risk timeframe, and making up the 18% average, would be approximately 1% oxide resource, 2% mixed, 8% sulphide and 6% mineralised waste

The low, medium and high risk mineralisation volumes are dealt with in the above described smoothing process when distinguishing between the different mineralisation styles and emulating mineable SMU parcels of ore. They are also considered when weighting and assessing the applicable allowances for unplanned mining losses over the life of mine (i.e. beyond the current five year budgeting timeframe).

Figure 15-4 Metal loss risk across the LOM timeframe in relation to Mineral Resource tonnage



15.4 Pit optimisation

Conventional Whittle Four-X software was used to produce a reference pit shell for the Main and North West deposits conformable with the existing extents of open pit mining and the practical cutbacks required to achieve the planned production expansion. The combined deposit optimisation also included the South East Dome deposit. This approach was comparable to that adopted for the 2020 mine planning, in so far as the reference optimisation shell was used as **a guide to refining cutback designs** rather than determining the ultimate pit limits.

The optimisation accounted for recoveries to copper metal in concentrate, as determined for variable Cu and fixed Au process recovery relationships. Recovery to anode and the metal costs associated with processing concentrate at the KCS were also taken into account.

15.4.1 Optimisation input parameters

The pit optimisation was completed using metal prices and operating costs reflective of those prevailing at the time. The impact of more recent, higher metal prices is addressed in the following commentary.

Pit slope parameters

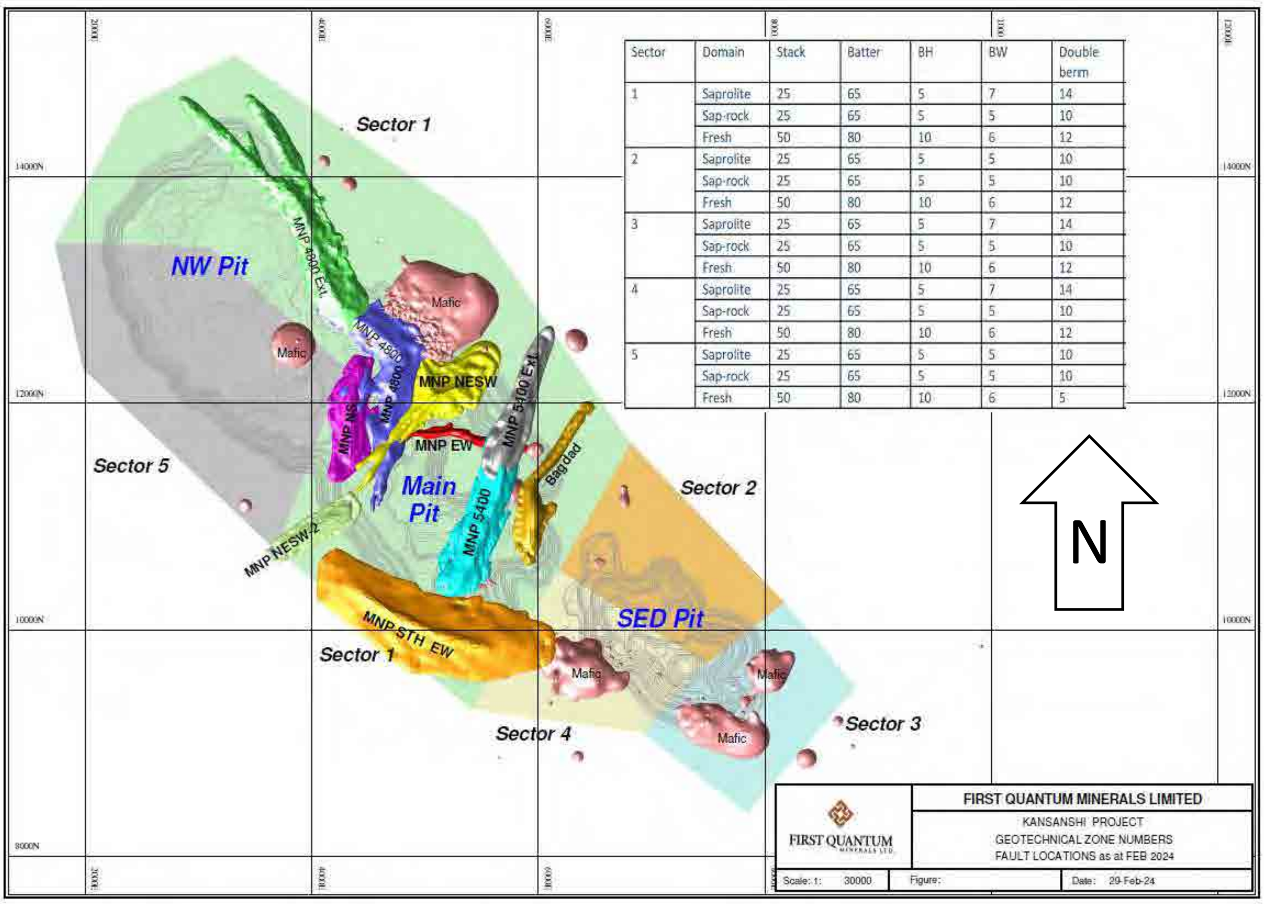
Pit optimisation input included overall slope design angles (inter-ramp angles) as listed in Table 15-2. The relevant pit design sectors are shown in Figure 15-5, whilst the slope specification by domain relates to the dividing horizons between weathered (saprolite and saprock) and fresh rock as interpreted from the geological model. These specifications were current as at December 2023.

Table 15-2 Overall pit slope angles (inter-ramp angles) for optimisation input 2023

Pit	Sector	Domain	Depth extent	Stack height (m)	Batter angle (degrees)	Bench height (m)	Berm width (m)	Double berm requirement	Inter-ramp angle (degrees)
Main & NW	**1**	Saprolite	as per the geological model	25	65	5	7	at 25 m height	28
		Saprock		25	65	5	5	at 25 m height	34
		Fresh rock		50	80	10	6	at 50 m height	52
SE Dome	**2**	Saprolite	as per the geological model	25	65	5	5	at 25 m height	34
		Saprock		25	65	5	5	at 25 m height	34
		Fresh rock		50	80	10	6	at 50 m height	52
	3	Saprolite	as per the geological model	25	65	5	7	at 25 m height	28
		Saprock		25	65	5	5	at 25 m height	34
		Fresh rock		50	80	10	6	at 50 m height	52
	4	Saprolite	as per the geological model	25	65	5	7	at 25 m height	28
		Saprock		25	65	5	5	at 25 m height	34
		Fresh rock		50	80	10	6	at 50 m height	52
Main & NW	**5**	Saprolite	as per the geological model	25	65	5	5	at 25 m height	34
		Saprock		25	65	5	5	at 25 m height	34
		Fresh rock		50	80	10	6	at 50 m height	52

Geotechnical information in relation to the updated slope design parameters is outlined in in Item 16.13.

Figure 15-5 Kansanshi pit slope design sectors, 2023 (source: FQM)



Metal prices

The optimisation inputs for long term metal prices were as follows:

- copper = \$3.50/lb (\$7,717/t)
- gold = \$1,805/oz

Metal recoveries

The input processing recovery projections were the same as those listed in Item 13 (Table 15-3).

Table 15-3 Kansanshi process recovery relationships 2023

Sulphide Ore Flotation	**%Cu rec = 28*ASCu/TCu + 92.5*AICu/TCu + 2.1*(TCu - 0.82)**
Mixed Ore Flotation	%Cu rec = 20*ASCu/TCu + 88*AICu/TCu + 8*(TCu - 0.94)
Mixed Ore Float-Leach	%Cu rec = 20*ASCu/TCu + 88*AICu/TCu + 8*(TCu – 0.94) + 0.15*% MFT to leach
Oxide Ore Float	%Cu rec = 11*ASCu/TCu + 54*AICu/TCu + 1.95*(TCu - 0.82)
Oxide Ore Float-Leach	%Cu rec = 86*ASCu/TCu + 65*AICu/TCu + 7.21*(TCu - 0.95)

Mining costs

Variable mining costs comprising drill, blast, load and haul unit costs, were adopted for the optimisation. This was an update on the varying mining costs adopted for the optimisation described in the 2020 Technical Report (FQM, September 2020). The overall average mining costs for optimisation were as follows:

- average waste mining cost = $6.08/bcm
- average ore mining cost = $6.13/bcm

Item 21 provides information on the estimation of ore and waste mining costs for the optimisation and for subsequent cashflow modelling purposes.

Operating costs

The process operating costs for each circuit were updated since the 2020 Technical Report, to yield the overall average unit costs listed in Table 15-4.

Table 15-4 Kansanshi average process operating costs for pit optimisation

Circuit	Units	Fixed plus variable	Other direct	Total costs
Oxide (incl. SX)	$/t processed	$7.70	$2.49	$10.19
Mixed	$/t processed	$4.75	$2.46	$7.22
Sulphide S2	$/t processed	$4.84	$2.51	$7.35
Sulphide S3	$/t processed	$4.70	$2.38	$7.08

Item 21 provides further information on the process operating cost breakdown.

Metal costs

In addition to royalties calculated on a payable metal sold basis, input metal costs for the Kansanshi product streams comprise:

- cathode metal production costs (EW consumables and engineering)
- cathode freight costs
- KCS smelting costs
- anode freight costs
- anode refining costs
- gold refining costs

Item 21 provides an explanation of the derivation of the metal costs used for pit optimisation input and as itemised in Table 15-5.

Table 15-5 Kansanshi metal costs for pit optimisation, excluding royalties

Transport, smelting and refining charges	Units	$/unit
Cathode EW costs		
Total cathode cost	$/lb Cu	0.30
Smelting/refining costs		
Total smelting/refining cost	$/lb Cu	0.32
Gold refining charge	$/lb oz	5.00

Mining dilution and recovery factors

The optimisation process included an allowance for "unplanned" mining dilution and ore losses, i.e. those occurring after the bench mark out and attributable to loading and hauling ore and waste from the pit to the correct, or inadvertently, to the incorrect destination. The overall, longer term, applied unplanned mining dilution and mining recovery factors were 101.5% and 98.5%, respectively. These factors are consistent with the 2020 Technical Report optimisation inputs.

Optimisation inputs summary

Table 15-6 lists updated pit optimisation parameters applicable after adopting the mining, processing, general and administration (G&A) and metal costs as per the information and tables above. The inputs from the 2020 Technical Report are also listed in Table 15-6 for comparison.

Marginal cut-off grades

Whittle optimisation software uses the following simplified formula to calculate the marginal cut-off grade as listed in Table 15-6:

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

The tabled equivalent cut-off grades in Table 15-7 are indicative overall average grades; the actual cut-off grade for each model block varies due to the variable process recovery equations.

Relative to the optimisation inputs in 2020 however, and reflecting averages across all circuits, these two tables indicate:

- adoption of higher long term metal prices (i.e. 17% higher for copper and 50% higher for gold)
- an average 5% reduction in copper recovery
- an average 50% reduction in gold recovery
- an approximate 18% reduction in ore and waste mining costs
- an approximate 17% increase in processing (treatment) plus G&A costs
- a 34% increase in metal costs, largely due to the inclusion of a ZCCM royalty
- whilst the marginal cut-off grades are quoted in copper equivalence terms, the contribution of gold to net return is negligible
- in 2020, without the gold contribution, the indicative marginal cut-off grade (across all circuits) was taken to be approximately 0.2% Cu
- with the increased metal price input, and the changes to recovery projections and to processing and metal costs, the indicative average cut-off grade across all circuits remains similar

Table 15-6 shows that a 6.9% royalty rate on copper was adopted as one of the 2024 optimisation inputs, whereas it was realised during subsequent cashflow modelling that the rate should be 6.0% (Item 22.1.2). Considering the context of the optimisation work, the impact of this variation is negligible in terms of the indicative, overall marginal cut-off grades listed in Table 15-7.

Table 15-6 Kansanshi pit optimisation inputs, 2020 and 2024

KANSANSHI OPTIMISATION	Units	2020 INPUTS				2024 INPUTS			
PROCESS		Float/leach		Float		Float/leach		Float	
Ore types		Non-primary		Primary		Non-primary		Primary	
		Leach	Mixed	Sulphide S2	Sulphide S3	Leach	Mixed	Sulphide S2	Sulphide S3
Smelter		**Kansanshi**				**Kansanshi**			
Optimisation Metal Price									
Copper price	$/lb	3.00	3.00	3.00	3.00	3.50	3.50	3.50	3.50
Copper price	$/tonne	6,614	6,614	6,614	6,614	7,716	7,716	7,716	7,716
Gold price	$/oz	1,200	1,200	1,200	1,200	1,805	1,805	1,805	1,805
Mining Parameters		Overall averages				Overall averages			
Mining Recovery	Factor	98.7%	98.7%	98.7%	98.7%	98.5%	98.5%	98.5%	98.5%
Dilution Factor	Factor	101.3%	101.3%	101.3%	101.3%	101.50%	101.5%	101.5%	101.5%
Metal Recovery Factors		Overall averages				Overall averages			
Smelter Recovery	%	97.0%	97.0%	97.0%	97.0%	97.8%	97.8%	97.8%	97.8%
Overall Average Cu Recovery	%	79.3%	72.0%	90.3%	89.4%	72.8%	67.7%	87.7%	87.7%
Overall Average Au Recovery	%	60.2%	68.5%	66.3%	66.3%	24.0%	32.0%	37.5%	37.5%
Mining Costs									
overall average waste cost	$/bcm mined	7.13	7.13	7.13	7.13	6.08	6.08	6.08	6.08
overall average ore cost	$/bcm mined	7.33	7.33	7.33	7.33	6.13	6.13	6.13	6.13
Treatment and G&A Costs									
SX Costs									
Sub-total SX	$/t process	0.97				0.96			
Process Costs									
Sub-total process	$/t process	5.96	4.24	6.03	5.42	6.73	4.75	4.84	4.70
Other Direct Costs									
Sub-total other direct	$/t process	2.08	2.08	2.09	0.00	2.49	2.46	2.51	2.38
G&A Costs									
Sub-total G&A	$/t process	1.05	1.05	0.80	0.73	1.55	1.55	1.55	1.55
Total Treatment	**$/t process**	**10.06**	**7.37**	**8.92**	**6.15**	**11.74**	**8.76**	**8.90**	**8.63**
Royalties									
Royalty rate - copper	%	7.5%	7.5%	7.5%	7.5%	6.9%	6.9%	6.9%	6.9%
Royalty rate - gold	%	6.0%	6.0%	6.0%	6.0%	6.0%	6.0%	6.0%	6.0%
ZCCM-IH royalty	%	n/a	n/a	n/a	n/a	3.1%	3.1%	3.1%	3.1%
Metal Costs Cathode Copper									
Total	**$/lb Cu**	**0.15**				**0.30**			
Copper Metal Costs									
Concentrate Grade	% Cu	24%	24%	24%	24%	23%	23%	23%	23%
Moisture Content	%	10%	10%	10%	10%	10%	10%	10%	10%
Kansanshi Smelter Treatment									
Treatment Cost (wet)	$/t conc								
Treatment Cost (dry)	$/t conc		88.89	88.89	88.89		89.57	89.57	89.57
subtotal	**$/t Cu**		**370.37**	**370.37**	**370.37**		**383.05**	**383.05**	**383.05**
subtotal	**$/lb Cu**		**0.17**	**0.17**	**0.17**		**0.17**	**0.17**	**0.17**
Anode Transport Costs									
Transport (dry, incl insurance)	$/t metal		183.64	183.64	183.64		270.46	270.46	270.46
subtotal	**$/t Cu**		**183.64**	**183.64**	**183.64**		**270.46**	**270.46**	**270.46**
subtotal	**$/lb Cu**		**0.08**	**0.08**	**0.08**		**0.12**	**0.12**	**0.12**
Refining Charges									
Charges	$/t metal		134.80	134.80	134.80		123.53	123.53	123.53
subtotal	**$/t Cu**		**134.80**	**134.80**	**134.80**		**123.53**	**123.53**	**123.53**
subtotal	**$/lb Cu**		**0.06**	**0.06**	**0.06**		**0.06**	**0.06**	**0.06**
Concentrate Shipped to Copper Belt									
subtotal	**$/t Cu**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**
subtotal	**$/lb Cu**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**	**0.00**
Royalty charge (gross)									
subtotal	**$/t Cu**	**496.04**	**496.04**	**496.04**	**496.04**	**771.04**	**771.04**	**771.04**	**771.04**
subtotal	**$/lb Cu**	**0.23**	**0.23**	**0.23**	**0.23**	**0.35**	**0.35**	**0.35**	**0.35**
Total Cu Metal Cost									
Total	**$/t Cu**	**496.04**	**1,184.85**	**1,184.85**	**1,184.85**	**771.04**	**1,548.08**	**1,548.08**	**1,548.08**
Total	**$/lb Cu**	**0.23**	**0.54**	**0.54**	**0.54**	**0.35**	**0.70**	**0.70**	**0.70**
Total	**$/10kg Cu**	**4.96**	**11.85**	**11.85**	**11.85**	**7.71**	**15.48**	**15.48**	**15.48**
Metal Costs Gold in Concentrate									
Gold refining charge (gold in anode)	$/oz	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00
Royalty charge (gross)	$/oz	72.00	72.00	72.00	72.00	164.26	164.26	164.26	164.26
Total	**$/oz**	**77.00**	**77.00**	**77.00**	**77.00**	**169.25**	**169.25**	**169.25**	**169.25**

Table 15-7 Kansanshi average marginal cut-off grades

KANSANSHI OPTIMISATION	Units	2020 INPUTS				2024 INPUTS			
PROCESS		Float/leach		Float		Float/leach		Float	
Ore types		Non-primary		Primary		Non-primary		Primary	
		Leach	Mixed	Sulphide S2	Sulphide S3	Leach	Mixed	Sulphide S2	Sulphide S3
Smelter		Kansanshi				Kansanshi			
Marginal Cut-Off Grade Calculation									
Process Cost	**$/t process**	**10.06**	**7.37**	**8.92**	**6.15**	**11.74**	**8.76**	**8.90**	**8.63**
Dilution Factor		1.013	1.013	1.013	1.013	1.015	1.015	1.015	1.015
Net Return (Metal price - Total metal cost)									
Cu net return (recovered)	$/lb	2.08	1.77	2.22	2.20	2.08	1.89	2.45	2.45
Au net return (recovered)	$/lb	9,859	11,218	10,858	10,858	5,725	7,633	8,945	8,945
CuEq net return (recovered)	$/lb	0.14	0.16	0.13	0.13	0.04	0.07	0.07	0.08
Sub-total net return (recovered)	$/lb	2.22	1.94	2.35	2.33	2.11	1.96	2.53	2.54
Sub-total net return (recovered)	$/10kg	49.00	42.68	51.82	51.36	46.55	43.24	55.73	55.90
Marginal Cut-off Grade, Direct Feed	**% Cu_{eq}**	**0.21**	**0.17**	**0.17**	**0.12**	**0.26**	**0.21**	**0.16**	**0.16**

15.4.2 Optimisation results

Table 15-8 lists the inventories from the generated sequence of optimisation shells. The nominal optimal pit shell is shown as the revenue factor 1.00 shell, yielding a total pit size of 3.3 Bt, and with a total combined plant feed of 974 Mt.

15.4.3 Optimisation sensitivity

Several pit optimisation input sensitivity analyses (Table 15-9) were carried out in order to separately test the impact of changes to the mining costs, to the processing and metal costs (combined) and the processing recovery. The reference base case for comparison is the revenue factor 1.00 shell.

To examine the impact of lower and higher prices, several other revenue factor pit shells were considered. The revenue factor 0.85 pit shell relates to a copper price of $3.00/lb (consistent with the 2020 Technical Report), the factor 1.10 shell relates to $3.85/lb, whilst the factor 1.15 shell relates to a copper price of $4.00/lb. In this last instance, the $4.00/lb copper price is comparable to the long term consensus price adopted in one of the cashflow model sensitivity analyses (Item 22).

A number of conclusions could be drawn from these sensitivity analyses, as follows:

- higher prices yield bigger pits with additional lower grade plant feed, but with additional waste that, overall, reduces the incremental value
- if mining costs were +10% higher there would be 6% less feed and significantly less waste, for an overall minimal value reduction
- an additional 5% of unplanned dilution would result in additional (diluted) plant feed, less waste mined, also for an overall minimal value reduction
- processing and selling costs at +10% could be a misleading result given that one is an operating cost and the other is a metal cost netted from revenue
- nevertheless, when the more sensitive processing costs are isolated, it is likely that a 10% cost increase alone, would not alter the overall average (across all circuits) marginal cut-off grade significantly
- given that the recovery projections are based on routinely updated equations, the processing recovery scenarios may be hypothetical

Table 15-8 Kansanshi pit optimisation shell inventories

Pit Number	Revenue Factor	Pit Size (Mt)	Waste Mined (Mt)	Strip Ratio	Oxide Feed (Mt)	Mixed Feed (Mt)	Sulphide Feed (Mt)	Total Plant Feed All Circuits (Mt)	(TCu %)	(Au g/t)	Contained Cu metal (kt)	Contained Au metal (koz)
1	0.50	337.2	203.5	1.5	18.6	32.0	83.0	133.7	1.21	0.21	1,617.3	902.4
2	0.55	547.0	347.8	1.7	24.7	42.0	132.5	199.2	1.06	0.19	2,111.3	1,216.7
3	0.60	705.7	451.3	1.8	31.4	51.9	171.1	254.3	0.97	0.17	2,467.2	1,390.1
4	0.65	988.5	652.3	1.9	40.6	63.5	232.1	336.2	0.88	0.16	2,958.2	1,729.2
5	0.70	1,286.1	873.3	2.1	49.8	74.3	288.7	412.7	0.82	0.15	3,384.4	1,990.3
6	0.75	1,662.0	1,148.3	2.2	60.7	87.9	365.0	513.7	0.76	0.14	3,904.0	2,312.0
7	0.80	1,830.4	1,250.7	2.2	69.4	97.3	413.0	579.7	0.72	0.13	4,173.9	2,422.9
8	0.85	2,006.6	1,360.9	2.1	78.7	107.9	459.1	645.7	0.68	0.13	4,390.7	2,698.6
9	0.90	2,179.0	1,970.4	2.6	86.8	116.0	504.1	706.8	0.65	0.12	4,594.2	2,726.8
10	0.95	2,705.7	2,680.6	2.9	104.1	129.9	593.3	826.3	0.61	0.12	5,040.2	3,187.7
11	**1.00**	**3,329.2**	**2,355.3**	**2.4**	**117.4**	**142.4**	**714.1**	**973.9**	**0.57**	**0.11**	**5,551.3**	**3,444.1**
12	1.05	3,652.7	2,583.6	2.4	129.7	152.3	787.0	1,069.0	0.55	0.11	5,879.6	3,780.5
13	1.10	3,786.3	2,649.3	2.3	139.6	160.7	836.3	1,137.0	0.53	0.11	6,026.2	4,021.0
14	1.15	4,051.6	2,829.4	2.3	152.6	171.0	898.6	1,222.2	0.51	0.10	6,233.5	3,929.4
15	1.20	4,110.3	2,841.5	2.2	161.4	176.8	930.5	1,268.8	0.50	0.10	6,343.8	4,078.9
16	1.25	4,143.7	2,838.1	2.2	169.9	182.1	953.5	1,305.6	0.49	0.10	6,397.2	4,197.2
17	1.30	4,164.7	2,827.9	2.1	178.3	187.3	971.3	1,336.8	0.48	0.10	6,416.7	4,297.8
18	1.35	4,197.1	2,828.8	2.1	186.7	191.9	989.7	1,368.3	0.47	0.10	6,431.0	4,399.0
19	1.40	4,936.1	3,448.2	2.3	203.4	206.3	1,078.3	1,488.0	0.46	0.10	6,844.6	4,783.7
20	1.45	6,136.4	4,486.6	2.7	216.5	213.5	1,219.8	1,649.8	0.44	0.09	7,259.1	4,773.6
21	1.50	6,657.3	4,912.9	2.8	225.6	218.9	1,299.9	1,744.3	0.43	0.09	7,500.5	5,047.0

Table 15-9 Kansanshi pit optimisation sensitivity analyses

Scenario	Pit Size (Mt)	Waste Mined (Mt)	Strip Ratio	Total Contained Plant Feed (Mt)	(%TCu)	Contained Cu metal (kt)	Contained Au metal (koz)	Average Value $M	Delta Feed (%)	Delta Contained Cu metal (%)	Delta Waste (%)	Delta Pit Size (%)	Delta Value (%)
Mining Costs -30%	4,095.7	2,984.1	2.7	1,111.6	0.54	6,002.6	3,931.1	**$16,070.9**	114%	108%	127%	123%	**122%**
Process & Selling Costs -10%	3,380.7	2,269.3	2.0	1,111.4	0.53	5,890.4	3,930.4	**$15,483.0**	114%	106%	96%	102%	**118%**
Mining Costs -20%	3,677.1	2,626.2	2.5	1,050.9	0.55	5,780.0	3,716.4	**$15,283.1**	108%	104%	112%	110%	**116%**
Process Recovery +5%	3,332.3	2,280.9	2.2	1,051.4	0.55	5,782.7	3,718.2	**$15,146.7**	108%	104%	97%	100%	**115%**
Mining Costs -10%	3,435.8	2,422.0	2.4	1,013.8	0.56	5,677.3	3,585.2	**$14,543.6**	104%	102%	103%	103%	**111%**
Base Case RF 1.0 ($3.50/lb Cu)	**3,329.2**	**2,355.3**	**2.4**	**973.9**	**0.57**	**5,551.3**		**$13,147.9**					
Mining Costs +10%	2,842.2	1,924.1	2.1	918.1	0.58	5,325.0	3,246.8	**$13,143.0**	94%	96%	82%	85%	**100%**
RF 1.10 ($3.85/lb Cu)	3,786.3	2,649.3	2.3	1,137.0	0.53	6,026.2	4,021.0	**$13,053.4**	117%	109%	112%	114%	**99%**
Mining Dilution +5%	3,089.2	2,106.2	2.1	983.0	0.55	5,406.5	3,476.3	**$12,986.1**	101%	97%	89%	93%	**99%**
RF 1.15 (~$4.00/lb Cu)	4,051.6	2,829.4	2.3	1,222.2	0.51	6,233.5	3,929.4	**$12,920.2**	125%	112%	120%	122%	**98%**
RF 0.85 (~$3.00/lb Cu)	2,006.6	1,360.9	2.1	645.7	0.68	4,390.7	2,698.6	**$12,685.6**	66%	79%	58%	60%	**96%**
Mining Costs +20%	2,635.1	1,753.9	2.0	881.2	0.58	5,111.0	3,116.3	**$12,533.2**	90%	92%	74%	79%	**95%**
Process & Selling Costs +10%	2,863.9	2,025.3	2.4	838.6	0.61	5,115.5	3,235.2	**$12,212.5**	86%	92%	86%	86%	**93%**
Mining Costs +30%	2,445.0	1,709.8	2.3	735.2	0.59	4,337.7	2,836.3	**$11,956.6**	75%	78%	73%	73%	**91%**
Process Recovery -5%	2,813.1	1,924.3	2.2	888.8	0.58	5,155.0	3,143.2	**$11,472.3**	91%	93%	82%	84%	**87%**
Mining Loss +5%	2,921.9	2,019.8	2.2	902.1	0.55	4,961.6	3,190.2	**$11,433.2**	93%	89%	86%	88%	**87%**

From Table 15-9 it can be concluded that the higher price optimisations do not necessarily equate to a larger pit with a better undiscounted cashflow value. On this basis it is considered that the adoption of an updated $3.50/lb longer term copper price is appropriate for Mineral Reserve reporting. The $4.02/lb copper (and $1,805/oz gold) in the cashflow model sensitivity analyses simply updates the revenue to current prices without changing the pit extents.

15.5 Detailed pit designs

Ultimate pit designs for Main, North West and South East Dome were produced using the selected pit shell in Table 15-8 as a notional guide. This shell envelops the phased pit designs produced for short and medium term site planning and reflects ultimate pit extents which are consistent with the practical cutbacks required to achieve the planned production expansion.

Table 15-10 lists the pit slope parameters used for detailed pit design, as specified and relevant at December 2023. Further geotechnical information in relation to pit slope design parameters is outlined in Item 16.

Table 15-10 Kansanshi pit slope design parameters, 2023

Pit	Sector	Domain	Depth extent	Stack height (m)	Batter angle (degrees)	Bench height (m)	Berm width (m)	Double berm requirement	Inter-ramp angle (degrees)
Main & NW	1	Saprolite	as per the geological model	25	65	5	7	at 25 m height	28
		Saprock		25	65	5	5	at 25 m height	34
		Fresh rock		50	80	10	6	at 50 m height	52
SE Dome	2	Saprolite	as per the geological model	25	65	5	5	at 25 m height	34
		Saprock		25	65	5	5	at 25 m height	34
		Fresh rock		50	80	10	6	at 50 m height	52
	3	Saprolite	as per the geological model	25	65	5	7	at 25 m height	28
		Saprock		25	65	5	5	at 25 m height	34
		Fresh rock		50	80	10	6	at 50 m height	52
	4	Saprolite	as per the geological model	25	65	5	7	at 25 m height	28
		Saprock		25	65	5	5	at 25 m height	34
		Fresh rock		50	80	10	6	at 50 m height	52
Main & NW	5	Saprolite	as per the geological model	25	65	5	5	at 25 m height	34
		Saprock		25	65	5	5	at 25 m height	34
		Fresh rock		50	80	10	6	at 50 m height	52

15.5.1 Other design and planning parameters

The following parameters relate to the design of pit slopes and ramps:

- benches (interval between berms) are mined to a height of 5 m or 10 m in ore and waste
- double width berms are designed at every stack height interval, i.e. every 25 m vertically for 5 m bench heights and every 50 m vertically for 10 m bench heights
- truck ramp operating width = 5 x truck width, plus additional allowances for a safety bund and drainage channel(s)
- total truck ramp width providing for trolley assisted haulage = 50 m for three lane ramps and 42.1 m for two lane ramps (refer to Figure 16-10 and Figure 16-11)
- maximum haul ramp gradient = 1 : 10, which is approximately 6°

15.5.2 Staged and ultimate pit designs

Figure 15-6 shows the starting point for the updated ultimate design (and production scheduling) process. This figure shows:

- the extent of mining, ore stockpiling and waste dumping as at the end of December 2023
- the crest of the current Mineral Reserve pit limits, marked as a green coloured perimeter

Figure 15-7 shows the updated ultimate design pit for the Main and North West Pits, whilst Figure 15-8 shows the updated ultimate design for the South East Dome Pit.

Figure 15-6 Kansanshi Operations site plan, December 2023 (source: FQM)

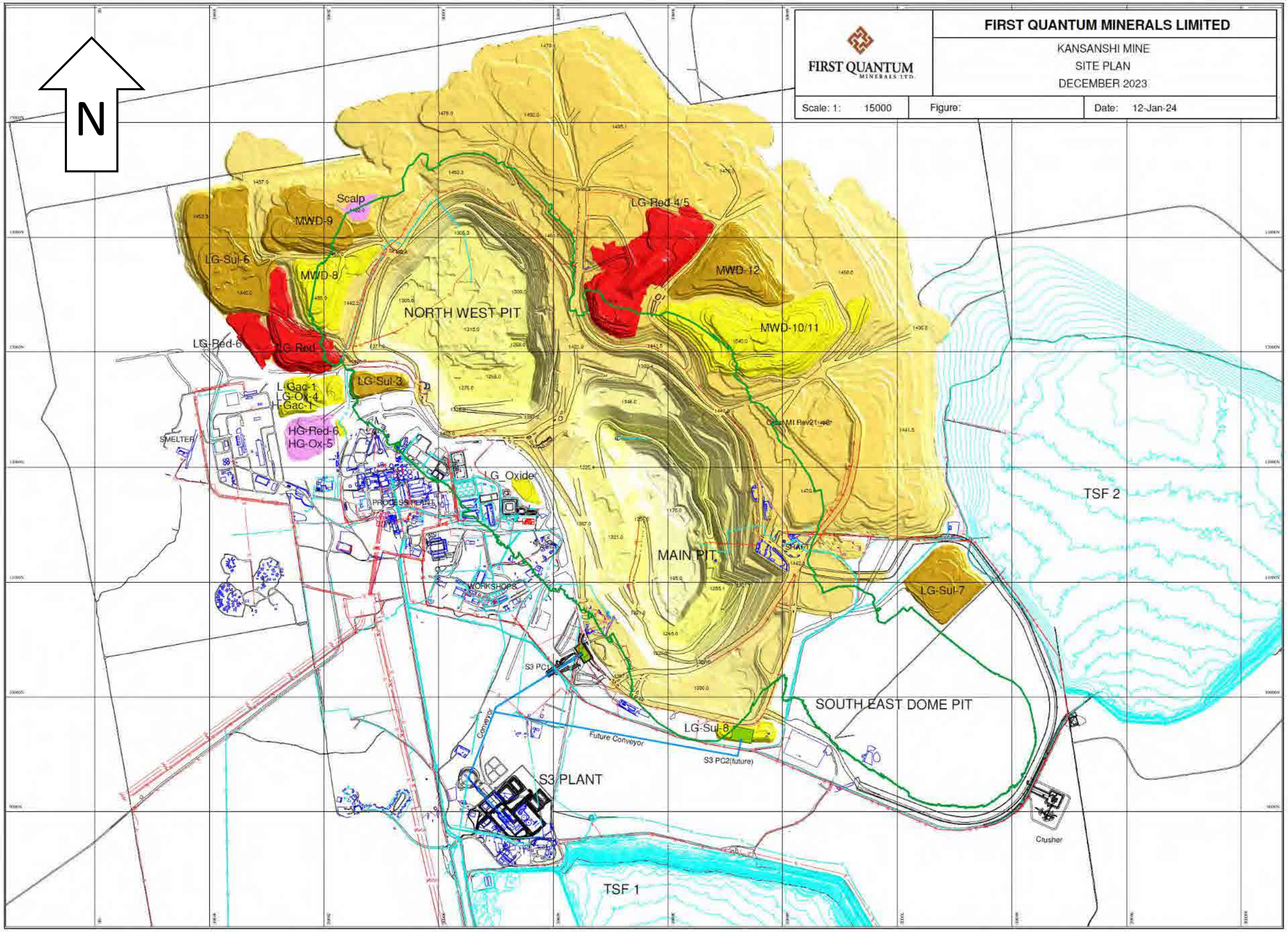


Figure 15-7 Main and North West ultimate pit designs, December 2023 (source: FQM)

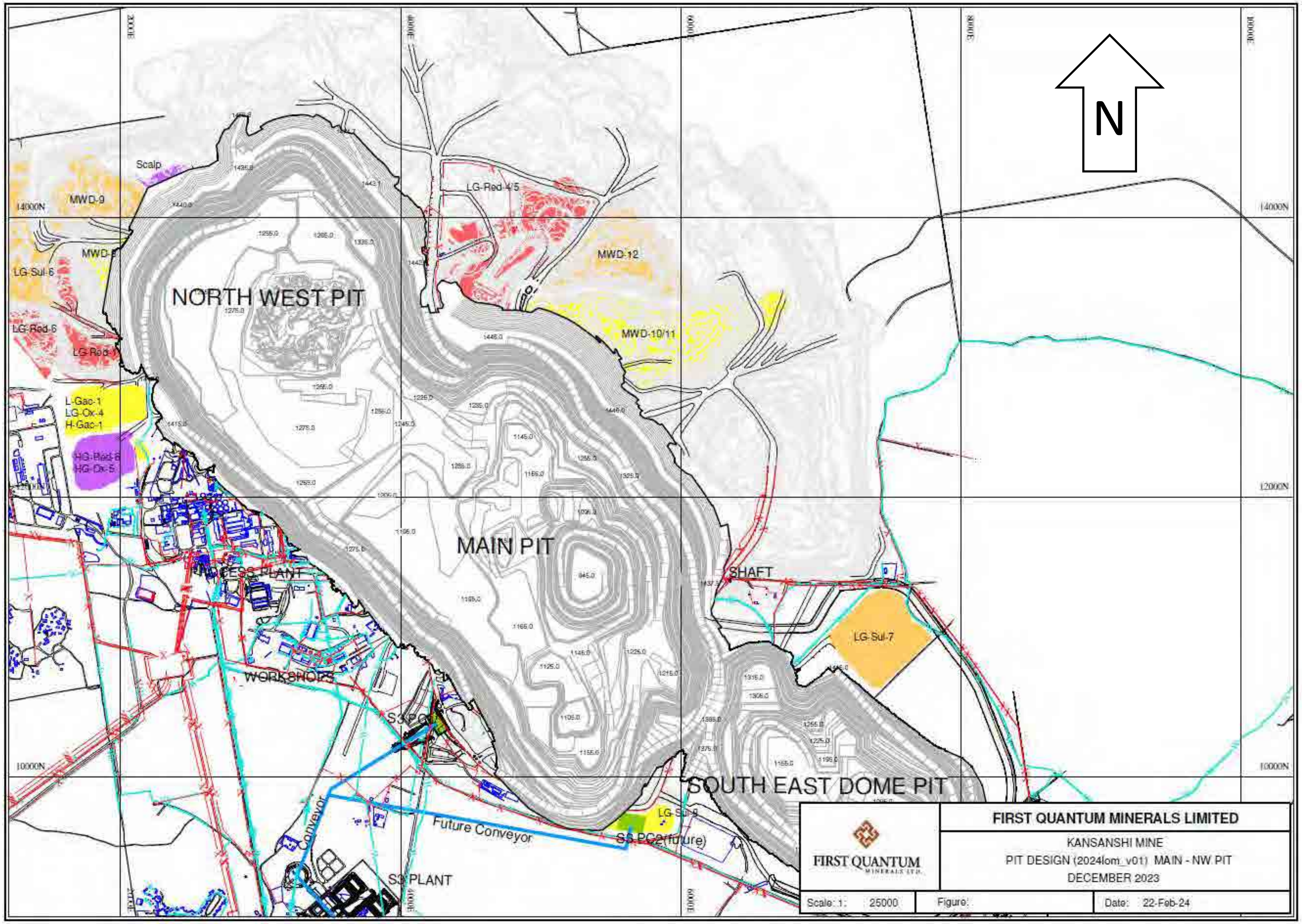


Figure 15-8 South East Dome ultimate pit design, December 2023 (source: FQM)

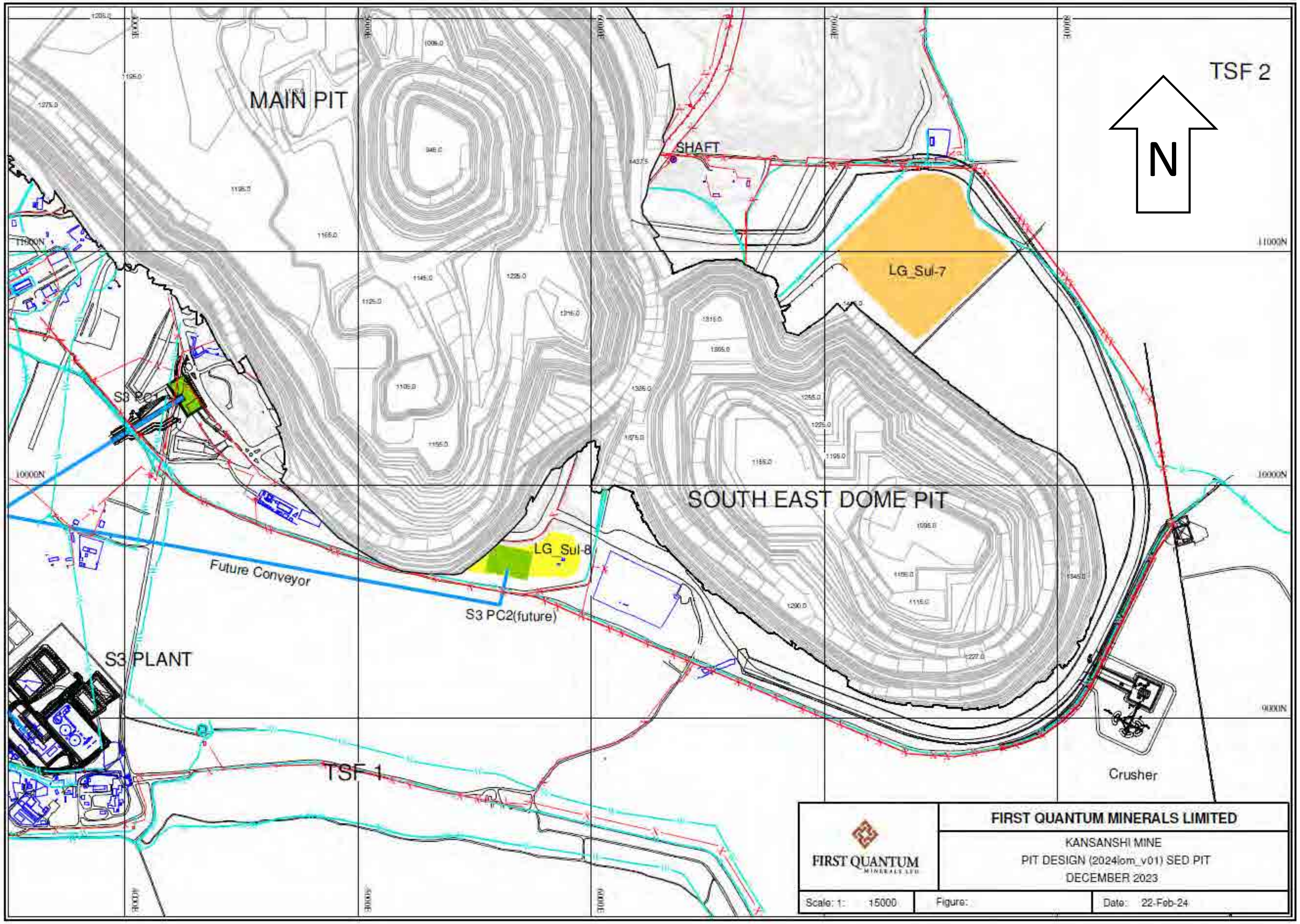
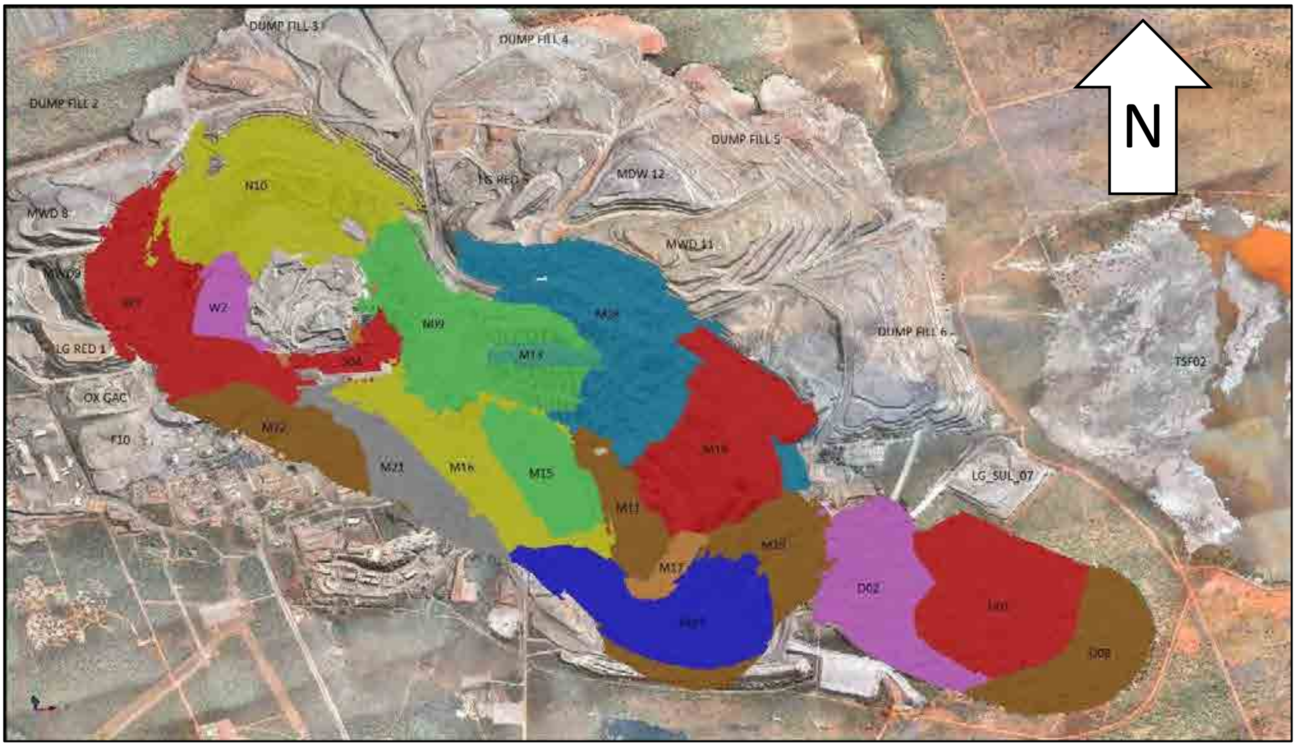


Figure 15-9 shows the nomenclature for the cutback phases (or stages) leading out to the ultimate pit extents.

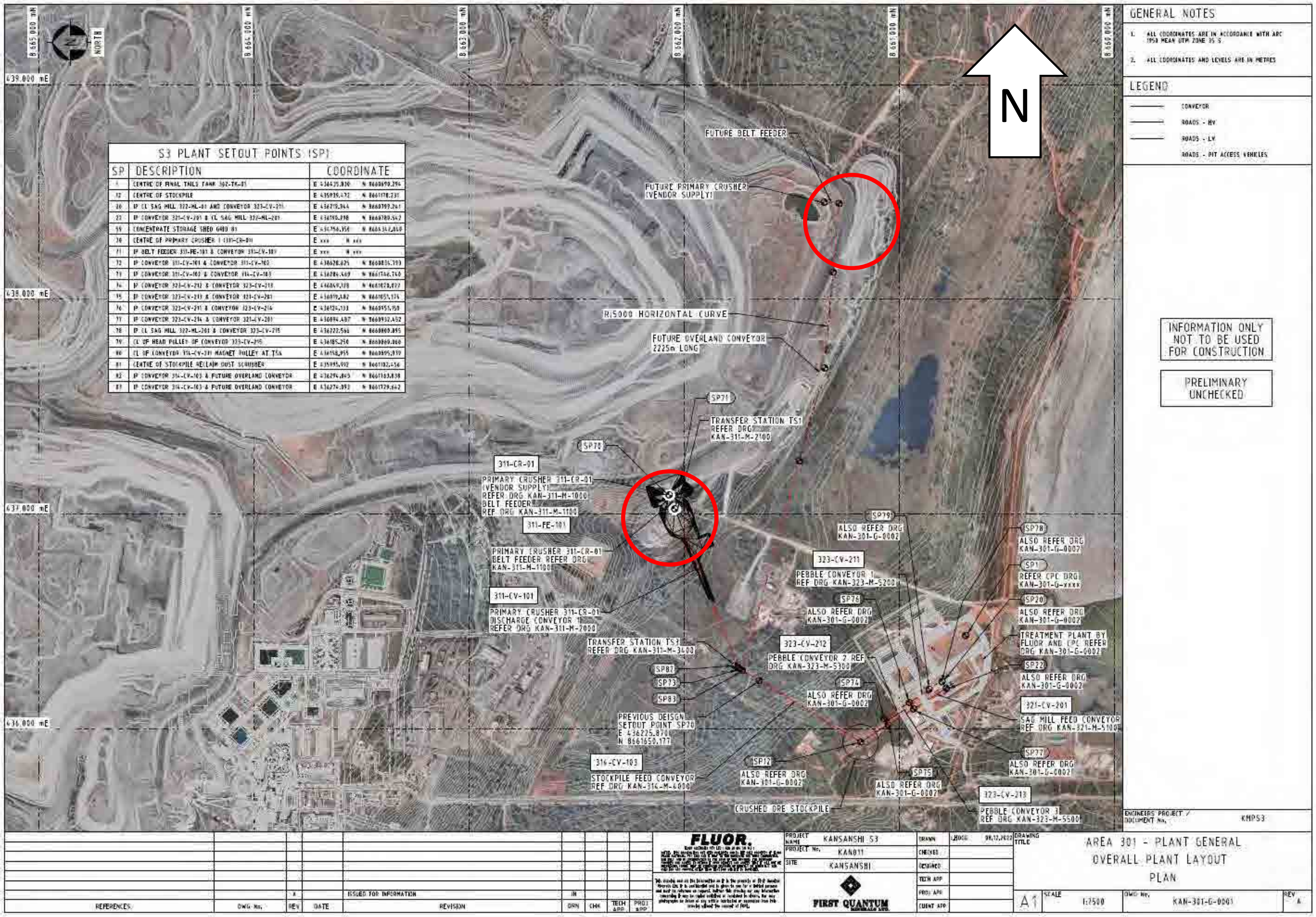
Figure 15-9 Nomenclature for pit cutback phases (source: FQM)



15.5.3 Primary crusher locations

The proposed locations of primary crushers serving the S3 expansion have changed since those depicted in the 2020 Technical Report. The positioning has been adjusted to suit the updated Mineral Reserve pits (Figure 15-10), and strictly speaking, these are now pit-rim crushers rather than in-pit crushers.

Figure 15-10 Kansanshi Main and South East Dome Pits, proposed primary crusher locations (source: Fluor)



15.6 Mineral Reserve statement

As at 31st December 2023, the Proven and Probable Mineral Reserve, inclusive of stockpile inventory, is estimated as 1,104.8 million tonnes at 0.54%TCu.

This estimate is within the Measured and Indicated Mineral Resource estimate reported in Table 14-36. A breakdown by pit and classification is provided in Table 15-11.

Table 15-11 Kansanshi Pit Mineral Reserve statement, at 31st December 2023

MINERAL RESERVE AT 31st DECEMBER 2023 (AT $3.50/lb Cu AND $1805/ounce Au)					
Oxide Ore					
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
MAIN & NW	Proven	118.7	0.53	0.25	0.09
	Probable	60.0	0.52	0.23	0.11
	Total P+P	**178.7**	**0.52**	**0.24**	**0.10**
SE DOME	Proven	15.3	0.43	0.15	0.07
	Probable	16.5	0.35	0.12	0.06
	Total P+P	**31.9**	**0.39**	**0.13**	**0.07**
TOTAL	**Proven**	**134.1**	**0.51**	**0.24**	**0.09**
	Probable	**76.5**	**0.48**	**0.21**	**0.10**
	Total P+P	**210.6**	**0.50**	**0.23**	**0.09**
Mixed Ore					
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
MAIN & NW	Proven	41.6	0.86	0.23	0.18
	Probable	29.1	0.86	0.22	0.16
	Total P+P	**70.6**	**0.86**	**0.22**	**0.17**
SE DOME	Proven	2.9	0.54	0.14	0.09
	Probable	8.0	0.66	0.17	0.10
	Total P+P	**10.9**	**0.63**	**0.16**	**0.09**
TOTAL	**Proven**	**44.4**	**0.84**	**0.22**	**0.17**
	Probable	**37.1**	**0.82**	**0.21**	**0.15**
	Total P+P	**81.5**	**0.83**	**0.21**	**0.16**
Sulphide Ore					
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
MAIN & NW	Proven	175.6	0.59	0.02	0.12
	Probable	297.0	0.51	0.02	0.12
	Total P+P	**472.6**	**0.54**	**0.02**	**0.12**
SE DOME	Proven	86.1	0.62	0.02	0.12
	Probable	84.4	0.51	0.01	0.09
	Total P+P	**170.5**	**0.57**	**0.02**	**0.11**
TOTAL	**Proven**	**261.7**	**0.60**	**0.02**	**0.12**
	Probable	**381.4**	**0.51**	**0.02**	**0.11**
	Total P+P	**643.1**	**0.55**	**0.02**	**0.12**
Total Ore					
Pit	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
MAIN & NW	Proven	335.9	0.60	0.13	0.12
	Probable	386.1	0.54	0.07	0.12
	Total P+P	**722.0**	**0.57**	**0.10**	**0.12**
SE DOME	Proven	104.3	0.59	0.04	0.11
	Probable	109.0	0.50	0.04	0.09
	Total P+P	**213.2**	**0.54**	**0.04**	**0.10**
TOTAL	**Proven**	**440.2**	**0.60**	**0.11**	**0.12**
	Probable	**495.0**	**0.53**	**0.06**	**0.11**
	Total P+P	**935.2**	**0.56**	**0.08**	**0.11**

Table 15-12 lists the additional Mineral Reserve inventory that was available on current stockpiles, as 31st December 2023.

Table 15-12 Kansanshi Stockpile Mineral Reserve statement, at 31st December 2023

MINERAL RESERVE AT 31st DECEMBER 2023 (AT $3.50/lb Cu AND $1805/ounce Au)					
Oxide Ore					
S/piles	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
TOTAL	Proven	-	-	-	-
	Probable	54.1	0.30	0.11	-
	Total P+P	**54.1**	**0.30**	**0.11**	**-**
Mixed Ore					
S/piles	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
TOTAL	Proven	-	-	-	-
	Probable	45.7	0.57	0.18	-
	Total P+P	**45.7**	**0.57**	**0.18**	**-**
Sulphide Ore					
S/piles	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
TOTAL	Proven	-	-	-	-
	Probable	69.7	0.36	0.01	-
	Total P+P	**69.7**	**0.36**	**0.01**	**-**
Total Ore					
S/piles	**Class**	**Mtonnes**	**TCu (%)**	**ASCu (%)**	**Au (g/t)**
TOTAL	**Proven**	**-**	**-**	**-**	**-**
	Probable	**169.5**	**0.40**	**0.09**	**-**
	Total P+P	**169.5**	**0.40**	**0.09**	**-**

The reported Mineral Reserve is based on detailed pit designs guided by an optimisation in which the marginal (economic) cut-off grade reflects longer-term copper and gold price projections of $3.50/lb ($7,717/t) and $1,805/oz, respectively.

The mining production schedule for the design Mineral Reserve pits is described in Item 16.14.

15.7 Comparison with previous Mineral Reserve estimate

Table 15-13 lists the Mineral Reserve as reported in the Company's Annual Information Form (AIF), as at 31st December 2023 (FQM, March 2024), and relative to the Mineral Resource model pertaining to the 2020 Technical Report. The reported total pit inventory is 715.3 Mt at an average grade of 0.62%TCu, containing 4,424.5 kt of insitu copper metal, and reflective of 127.3 Mt of mining depletion since June 2020.

This table also shows the AIF reported pit inventory relative to the new Mineral Reserve pit inventory as listed in Table 15-11. An overall uplift of 31% and 19% is evident, respectively, for Mineral Reserve ore tonnes and insitu copper metal.

Accompanying this table is a set of waterfall charts in Figure 15-11 and

Figure 15-12, each depicting the step changes between old and new Mineral Reserve statement inventories. In respect of the combined Main and North West pit inventories:

- there is a 20% increase in ore tonnage as a result of Mineral Resource model updates, related to the inclusion of new drilling/assaying data, estimation changes and improvements, and also reclassification

- despite the 20% tonnage increase, the corresponding contained copper metal increase is 17%, attributable to the reduction in the estimated overall average copper grade and consistent with the commentary in Item 14.21
- reflecting the new pit design, there is an additional 5% increase in both ore tonnage and contained copper
- there is a further undifferentiated increase and reduction of 2% and -7%, in ore tonnage and contained metal, respectively

In respect of the South East Dome inventory:

- there is a 44% increase in ore tonnage as a result of Mineral Resource model updates, related to the inclusion of new drilling/assaying data, estimation changes and improvements, and also reclassification
- there is a corresponding 51% increase in contained copper metal, implying an increase in the estimated overall average copper grade
- reflecting the new pit design, there is a 5% reduction in both ore tonnage and contained copper
- there is a further undifferentiated increase and reduction of 5% and -6%, in ore tonnage and contained metal, respectively

Table 15-13 Comparison between the depleted Kansanshi Pit Mineral Reserve in respect of the 2020 Technical Report model and the current Mineral Reserve statement as at 31st December 2023

	(Mt)	(%TCu)	Cu (kt)
30th June 2020 Pit Inventory	**2020 TR Resource Model**		
Main & NW Pits	694.4	0.66	4,565.4
SED Pit	148.3	0.56	828.9
Total	**842.7**	**0.64**	**5,394.3**
Depletion to 31st December 2023	**2020 TR Resource Model**		
Main & NW Pits	127.3	0.76	969.8
SED Pit	0.0	0.00	0.0
Total	**127.3**	**0.76**	**969.8**
AIF for 2023 (FQM, March 2024)	**2020 TR Resource Model**		
Main & NW Pits	567.1	0.63	3,595.6
SED Pit	148.3	0.56	828.8
Total	**715.3**	**0.62**	**4,424.5**
31st December 2023 Pit Inventory	**2024 TR Resource Model**		
Main & NW Pits	722.0	0.57	4,106.5
SED Pit	213.2	0.54	1,159.8
Total	**935.2**	**0.56**	**5,266.3**
Uplift 2020 to 2023 Pit Inventory			
Main & NW Pits	**127%**	**90%**	**114%**
SED Pit	**144%**	**97%**	**140%**
Total	**131%**	**91%**	**119%**

The "Balance (RM2024)" values shown in Figure 15-1 and Figure 15-2 are arithmetic balances, which whilst not readily differentiated, are likely attributable to:

- modelled density and estimated grade differences between Mineral Resource model versions
- incorporation of planned and unplanned metal losses in the mine planning model, upon which the Mineral Reserve estimate is derived

Figure 15-11 Kansanshi Main and North West Pits, Mineral Reserve waterfall chart

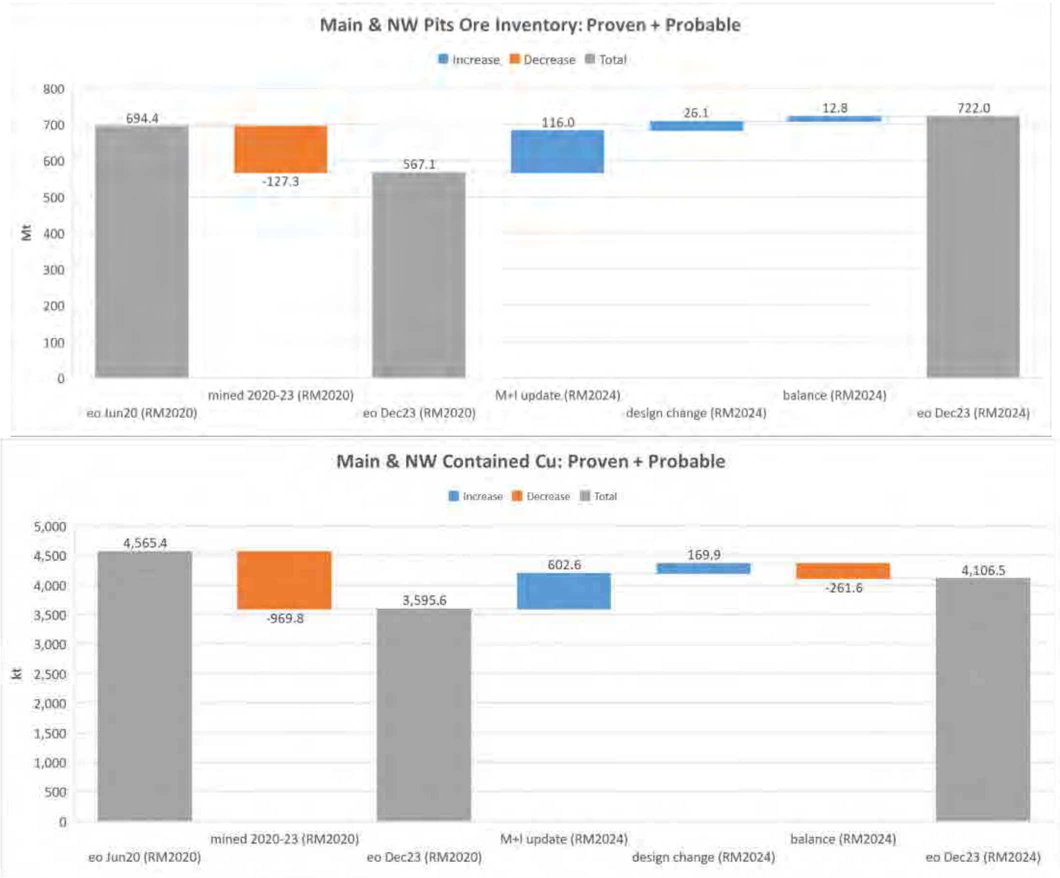
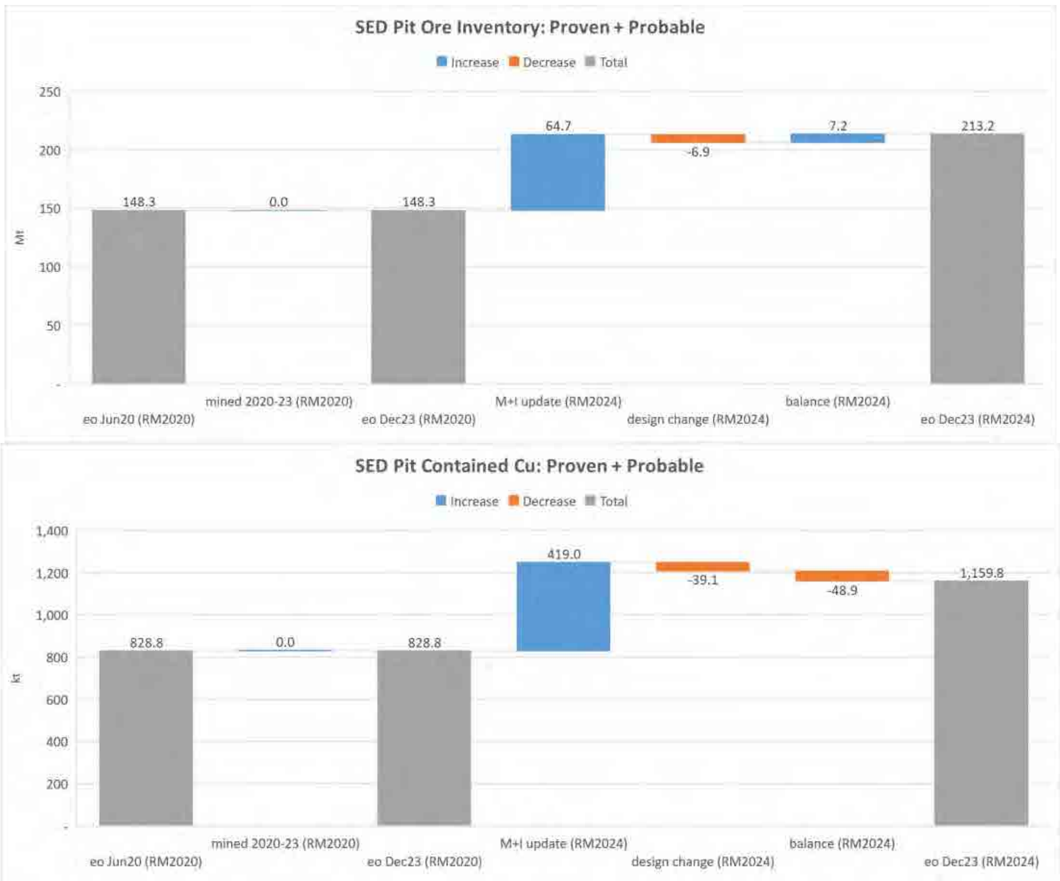


Figure 15-12 Kansanshi South East Dome Pit, Mineral Reserve waterfall chart



15.8 Factors and risks which may affect the Mineral Reserve estimate

Item 15.4.3 addressed pit optimisation sensitivity analyses on the impact of varying operating costs (mining and processing), metal costs and metallurgical recovery. Of these particular Mineral Reserve modifying factors, the obvious one that would have the largest impact on the Mineral Reserves estimate is a reduction in metallurgical recovery, as this would be analogous to a reduction in the plant feed grades.

As a mitigation of this risk, metallurgical recovery projections are continually updated based on operating performance, and in the case of future South East Dome plant feed, based on testwork and mineralogical comparisons against the plant feed currently being treated from the Main Pit (Item 13.3).

Mining cost increases by up to 10%, which are a possibility, would appear to reduce the pit size by about 15%, but without materially impacting Project value (at $3.50/lb copper price and ignoring capital costs). Processing cost increases to the same extent, and on the same optimisation copper price basis, would appear to reduce the plant feed inventory and pit size more significantly, and with a reduction in Project value by about 7%.

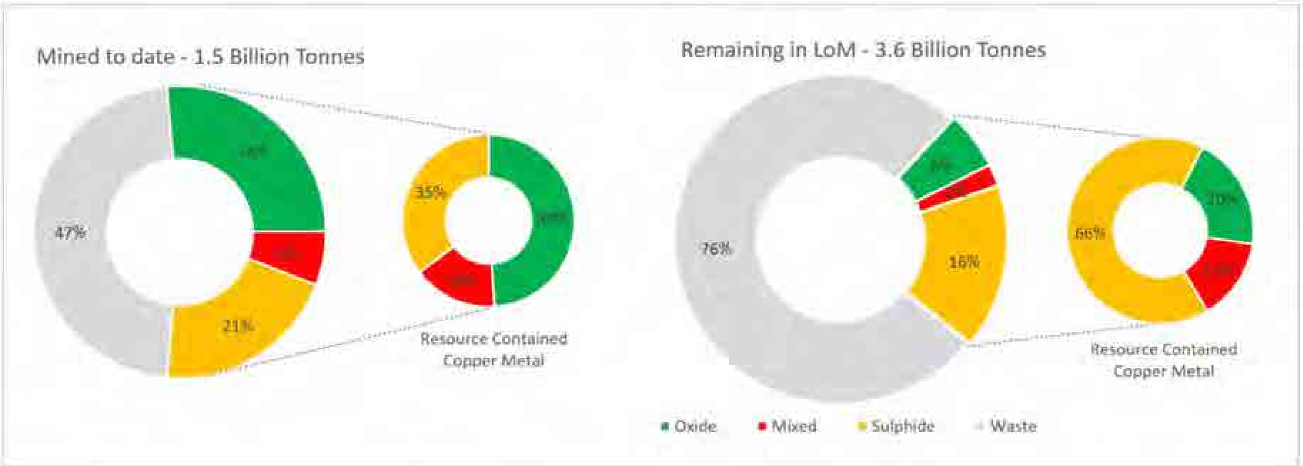
Increasing the copper price, in line with current consensus long term projections, would theoretically improve the Project value. However, owing primarily to the increasing waste strip ratio at depth, the higher revenue does not offset the cost of mining the additional waste to access incremental ore.

Further to the concluding remarks in Items 4.7 and 5.7, and to the extent known by the QP, there are no significant factors and risks that may affect access, title to, or the rights or ability to perform work on the property, to the extent that the Mineral Reserve estimate is materially affected.

ITEM 16 MINING METHODS

The Kansanshi ore production profile is changing and the near-surface high grade oxide ores, which have supported Kansanshi production to date are nearing depletion. This leaves a higher proportion of primary, lower grade sulphide ores at depth, with a high waste stripping ratio. To cater for this change, the S3 expansion comprises a new 27.5 Mtpa processing plant to treat the increasing tonnage of mined sulphide ore. Figure 16-1 illustrates the increased proportion of sulphide resource in the life of mine (LOM) plan, and also the increased strip ratio relative to what has been mined to date.

Figure 16-1 Kansanshi Mine, comparison of tonnes mined by ore (and waste) type; historical and remaining



This item provides information on the mining methods and operations as they exist currently, but with a broader coverage of proposed mining and operational aspects going forward. Included in this item is mine planning commentary relevant to the changed mining circumstances, such as mine design parameters, mining production schedules and primary mining equipment requirements.

16.1 Mining details

Open pit mining at Kansanshi features conventional drill and blast, shovel/excavator and truck mining techniques. Mining has proceeded from initial excavations in two pits (Main and North West Pits) through a sequence of cutbacks, which in the longer term will result in these pits merging. The South East Dome Pit will contribute to the longer-term production profile, but is unlikely to merge with the Main Pit. The cutbacks generally comprise wide benches of 200 m to 300 m width, providing several mining horizons from which to satisfy the feed requirements for multiple processing routes.

The current practice involves mining ore in low height flitches, using relatively small-scale equipment to facilitate mining selectivity. In general, ore is currently hauled to a ROM (run of mine) pad located immediately south of the North West Pit, and to a number of ore stockpiles, whereas waste is hauled to various dump locations.

To accommodate the increased processing requirements for the S3 expansion, the existing mining fleet will be expanded. Cutbacks for the Main and North West Pits have been designed to accommodate trolley-assist haulage routes, in addition to catering for proposed semi-mobile, pit-rim crusher locations and associated conveyor routes.

16.2 Mine site layout

The current layout of the Kansanshi mine site is shown in Figure 16-2. The main features shown are:

- the existing Main and North West Pits, and the location of the designed South East Dome Pit

- the extent of waste dumping around the northern periphery of the pits
- the location of the S3 plant site, relative to the existing process plant
- the updated Mineral Reserve life of mine extents (the green perimeter)

The current mining cutbacks in the Main and North West Pits are N09, D11, M11, M13, M15, M17, M18, D01 and D02 (refer to Figure 15-9). The current pit (as at 31st December 2023) measures 4,660 m along strike from north to south, is 1,160 m at its widest and 330 m beneath natural surface topography, at its deepest elevation of 1,195 mRL.

The LOM pit will measure 7,680 m along strike from north to south, is 2,585 m at its widest and 580 m beneath natural surface topography, at its deepest elevation of 945 mRL.

A considerable proportion of processing feed has been sourced from long term stockpiles, as these are typically of higher average grade than some of the ex-pit ore sources. Much of the material that has been sent to stockpiles has been oxide mineralised waste. Figure 16-3 shows the location of these stockpiles; additional information on the stockpiles and their respective end of December 2023 inventories is provided in Item 15.6.

16.3 Mining method and operations

Mining follows conventional drill and blast, shovel/excavator and truck mining practice. The sequence of mining activities is also conventional and is typically as follows:

- RC grade control drilling delineates the ore zones.
- A grade control model is developed from which blast limits and digging blocks are designed.
- Ore and waste blocks are blasted to design, according to layouts governed by varying hole patterns and powder factors to suit prevailing ground conditions and ore types.
- Electric and diesel/hydraulic shovels and excavators load the blasted rock into a fleet of 150 tonne to 220 tonne capacity haul trucks.
- Ore is hauled direct to a surface ROM (run of mine) pad or to stockpiles, whilst waste is hauled to surface dump tip heads.
- Trolley assisted haulage for diesel electric trucks is currently in use, and it is envisaged that the trolley assist network will be expanded over the LOM to provide extensive routes on pit and waste dump ramps, as well as selected flat haul sections, to maximise the productivity and cost benefits as the pit becomes progressively deeper.

The existing mining fleet and flitch mining approach will continue to be used in the currently active Main and North West Pit cutbacks, most of which will be completed by the end of 2025. All future cutbacks in these pits and at South East Dome are planned to be mined using larger equipment and single-pass benching to be able to move the volumes at the required pace to access sulphide ore and sustain the mill feed.

From 2027 onwards there will be substantial volumes of waste encountered, without adjoining ore contacts. This emphasises the importance of the bulk mining approach and a desire to expand the trolley-assisted haulage network over time.

Waste dumping capacity has become a constraint in recent times and to that end, and to avoid longer waste hauls in the future, space within the existing dump footprints will become available when overlying ore stockpiles are reclaimed and depleted. As described in the 2020 Technical Report (FQM, September 2020), waste backfilling into the depleted North West Pit remains an important part of the proposed mining plan.

Figure 16-2 Kansanshi Mine, December 2023 (source: FQM)

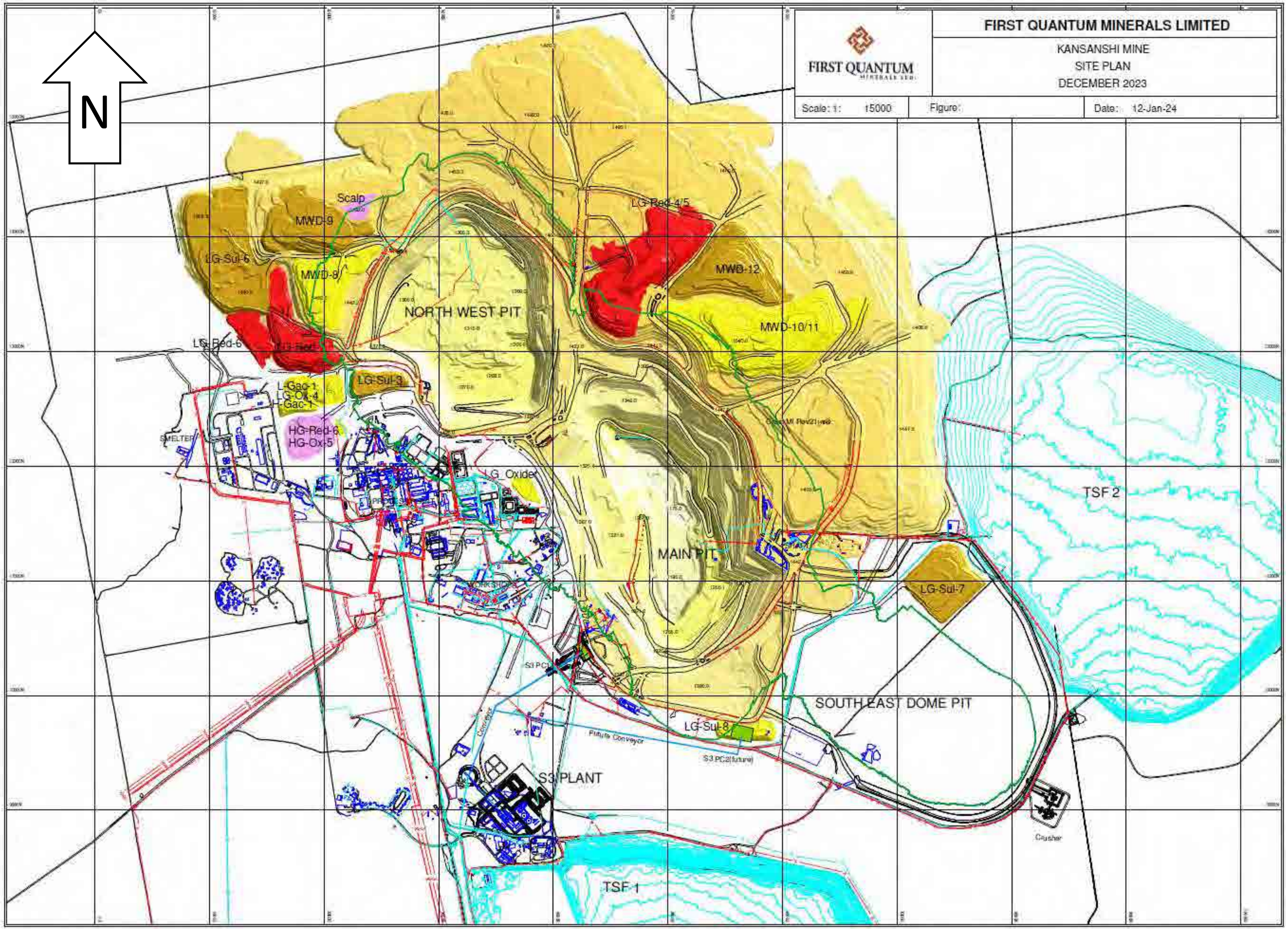
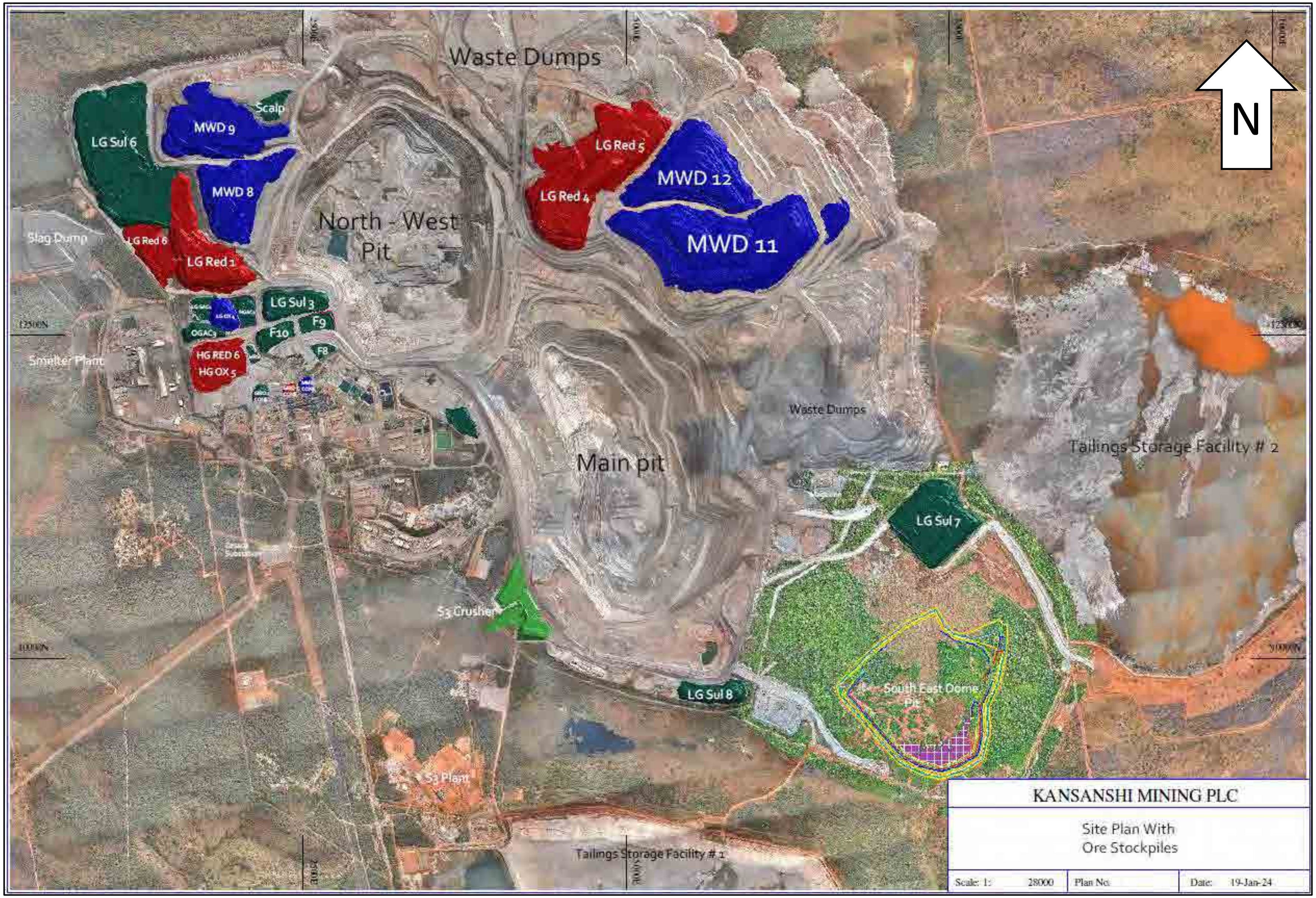


Figure 16-3 Kansanshi Surface Ore Stockpiles, December 2023 (source: FQM)



The green perimeter shown in Figure 16-2 is the crest position of the designed ultimate pit. Existing infrastructure will need to be removed from 2040 at the cessation of oxide, mixed and S2 plant processing. This will enable longer term Main Pit and North West Pit phase cut backs to proceed.

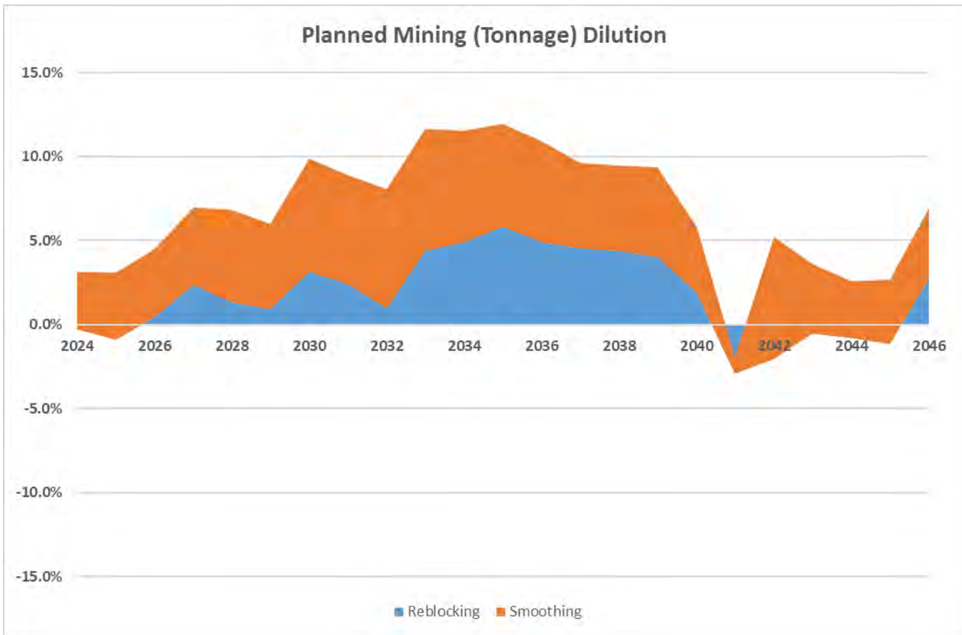
16.4 Mining dilution and losses

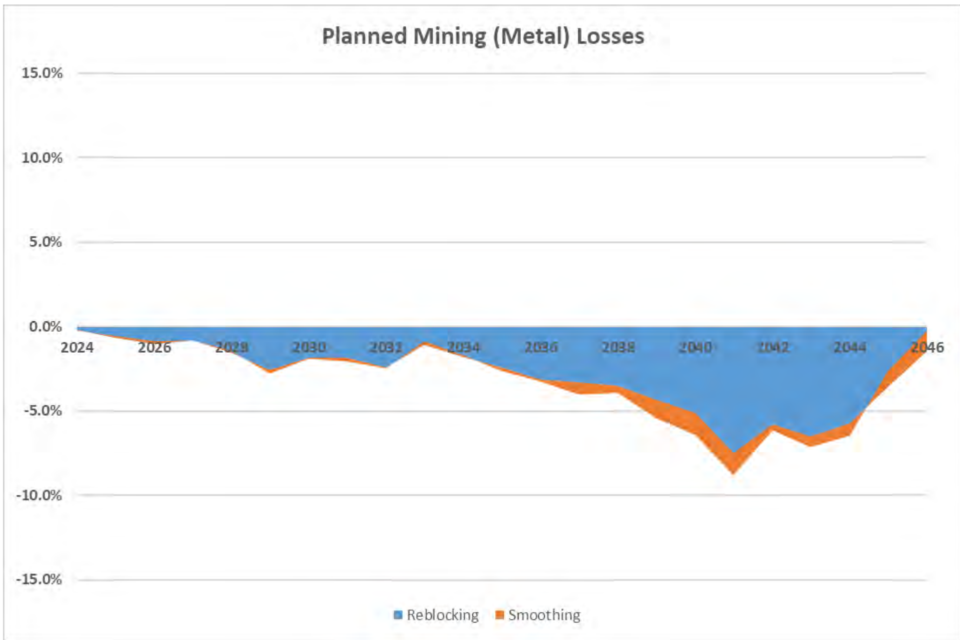
16.4.1 Planned mining dilution and losses

In the discussion on developing the mine planning model for the Mineral Reserve estimate, Item 15.3 described a two-step process of model reblocking and smoothing, the combined effect of which is the inclusion of "planned" mining dilution and losses into the model used for production scheduling and Reserves estimation.

Due to the method of developing the mine planning model, those domains of each pit with predominantly vein style mineralisation typically will have a higher average planned dilution than those domains with predominantly stratiform mineralisation. Whilst the different styles of mineralisation will be mined concurrently throughout the life of mine, the average planned mining dilution and losses will be variable for each year depending on the mineralisation styles being mined in that year. This effect is shown graphically in Figure 16-4.

Figure 16-4 Modelled planned mining dilution and losses





16.4.2 Unplanned mining dilution and losses

A mathematical weighting process was adopted to assess the applicable and varying unplanned metal losses (attributable to mining dilution and recovery losses) to be applied over the course of the LOM, and these modifying factors were then included in the development of the mine planning model used for design and production scheduling.

By means of a matrix calculation, the process incorporated a metal loss applied to each of the cutbacks within the LOM pit design inventory to reflect a change in mining practices over time. Combined with this was the metal risk considerations that are described in Item 15.3.4, relevant to varying mineralisation styles. The loss adjustments were assigned on the following basis:

- the metal losses in each cutback were assigned in recognition of the transition of each from selective to bulk mining methods (refer to the adjustments listed vertically in Table 16-1):
  - certain cutbacks received a relatively high adjustment to reflect the selective mining practices in current cutbacks
  - relatively lower losses were assigned thereafter to reflect the transition to 10 m high mining benches, with expected improvements in mining practices, and in mining dilution and loss control processes
  - in particular, losses incurred due to blast movement, which is linked to mineralisation style and hence metal loss risk, i.e. losses are more likely in the higher risk vein style mineralisation
- a metal loss of 6% (high risk) to 0.5% (low risk) was also applied (refer to the adjustments listed horizontally in Table 16-1); these particular adjustments were reflective of:
  - losses incurred during ore loading and haulage, arrived at by calibrating the combined adjustments for each year of the LOM inventory, post 2028, against a reconciliation from 2023 dispatch records

The matrix of combined and resultant tonnes weighted metal loss adjustments are listed in Table 16-1. From this matrix approach, an estimated varying annual unplanned metal loss adjustment can be graphed as shown in

Figure 16-5, superimposed over the risk proportions of copper metal tonnage within the LOM plan. This figure shows an unplanned metal loss adjustment ranging from a high of 9.2% to 0.7% longer term.

Figure 16-5 also shows a much higher profile of unplanned metal losses for the budget period 2024 to 2028 (and trending through 2029), ranging from an additional 17.1% to 2% above that indicted from the matrix calculations. This is attributable to the alignment adjustments mentioned in Item 15.3.3, accounting for:

- conservative adjustments applied for budgeting purposes
- rehandling from long term stockpiles being a significant performance indicator in recent years, and continuing to be so during the 2024 to 2029 timeframe

Depletion of stockpiles over the next five years will have a bearing on the projected longer term trend of reduced unplanned losses. Furthermore, the current mill constrained circumstances will diminish as the expanded capacity of the S3 plant is realised.

Table 16-1 Combined adjustments to allow for unplanned copper metal losses

Cutback	Cutback metal loss (%)	High risk loss		Medium risk loss		Low risk loss		Combined and weighted total metal loss (%)	Comment
		Proportion (%)	Metal loss (%) 6.0	Proportion (%)	Metal loss (%) 1.5	Proportion (%)	Metal loss (%) 0.5		
Main Pit									
m11	7.3	3.9%	13.3	16.9%	8.9	79.2%	7.8	8.2	Current cutback - budget plan input
m13	10.8	0.0%	16.8	5.7%	12.4	94.3%	11.3	11.4	Current cutback - budget plan input
m15	14.7	1.5%	20.7	12.8%	16.2	85.7%	15.2	15.4	Current cutback - budget plan input
m16	0.5	7.7%	6.5	21.1%	2.0	71.2%	1.0	1.6	Progressive five year incremental improvement
m17	8.5	0.0%	14.5	30.8%	10.0	69.2%	9.0	9.3	Current cutback - budget plan input
m18	0.5	3.7%	6.5	12.7%	2.0	83.7%	1.0	1.3	Progressive five year incremental improvement
m19	0.5	20.5%	6.5	27.6%	2.0	51.8%	1.0	2.4	Progressive five year incremental improvement
m21	0.5	18.4%	6.5	30.0%	2.0	51.6%	1.0	2.3	Progressive five year incremental improvement
m22	0.5	49.2%	6.5	29.6%	2.0	21.2%	1.0	4.0	Progressive five year incremental improvement
m23	0.5	22.1%	6.5	31.6%	2.0	46.3%	1.0	2.5	Progressive five year incremental improvement
m28	0.5	6.5%	6.5	21.3%	2.0	72.2%	1.0	1.6	Progressive five year incremental improvement
NW Pit									
n09	0.5	6.7%	6.5	24.9%	2.0	68.4%	1.0	1.6	Progressive five year incremental improvement
n10	0.5	36.6%	6.5	35.0%	2.0	28.4%	1.0	3.4	Progressive five year incremental improvement
s04	10.3	6.0%	16.3	40.3%	11.9	53.8%	10.8	11.6	Current cutback - budget plan input
w02	7.0	49.8%	13.0	49.0%	8.5	1.2%	7.5	10.8	Current cutback - budget plan input
w03	0.5	38.6%	6.5	35.7%	2.0	25.7%	1.0	3.5	Progressive five year incremental improvement
SED Pit									
d01	0.5	18.3%	6.5	49.9%	2.0	31.8%	1.0	2.5	Progressive five year incremental improvement
d02	0.5	42.9%	6.5	36.3%	2.0	20.8%	1.0	3.7	Progressive five year incremental improvement
d03	0.5	39.7%	6.5	40.0%	2.0	20.3%	1.0	3.6	Progressive five year incremental improvement

Figure 16-5 Graph of varying unplanned mining metal losses, over time, and relative to cutback and metal risk adjustments

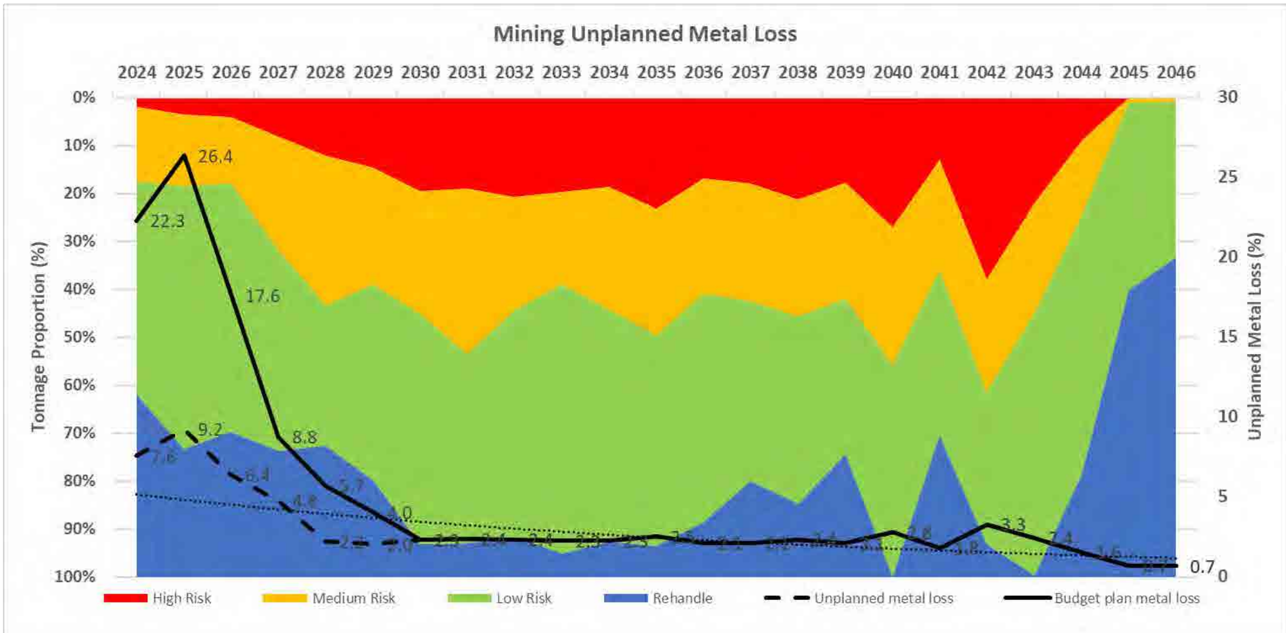
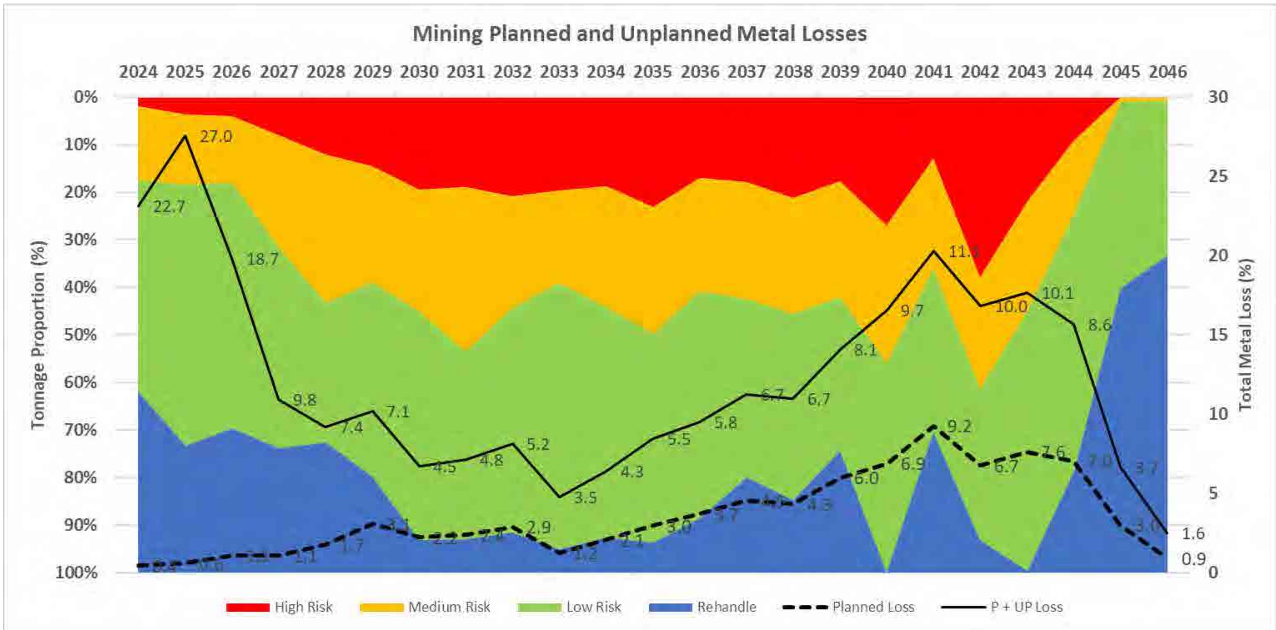


Figure 16-6 is a graph showing the combined annual trend of planned and unplanned metal losses over the LOM timeframe.

- in the period 2024 to 2028, reflecting the current budget period, the combined metal loss adjustment ranges from a high of 27.0% to 7.4%:
  - which reflects a high risk proportion of only 6% over this period, a medium risk proportion of 20%, and a low risk proportion of 44% (rehandle represents 30% over this period)
  - and which also reflects ~43% of the mined tonnage emanating from the current M15 (Main Pit) cutback, and ~33% of the mined tonnage from the two initial South East Dome Pit cutbacks
- after 2029, the combined metal loss adjustments trend downwards from 7.1% to 3.5% in 2033:
  - which reflects a high risk proportion increasing to 19% over this period, a medium risk proportion of 26%, and a low risk proportion of 46% (rehandle reduces and represents 9% over this period)
  - and which also reflects substantial mined tonnage emanating from the future bulk mined M16 and M18 (Main Pit) cutbacks, and mined tonnage coming from the third of the South East Dome Pit cutbacks
- whilst the unplanned losses essentially trend at a relatively low level from 2033, the underlying planned losses are modelled with an increasing profile:
  - to reflect the mining of the third cutback at South East Dome, together with substantial tonnages mined from the m16, m19 and m28 cutbacks in Main Pit
  - to reflect a trend towards a higher metal risk proportion between 2040 and 2043

Figure 16-6 Graph of varying planned and unplanned mining metal losses, over time



16.4.3 Production tracking records

A review of available production tracking records was carried out to assess the magnitude of unplanned metal losses occurring across current mining areas during 2023.

Table 16-2 summarises the reconciliation steps determined from these records, in terms of the ore types within the 2023 post-blast polygon mark-out *vs* the ex-pit FMS (fleet management system) dispatch records for those same material types[7].

When comparing the records across individual ore types (i.e. codes), this table does shows variable metal losses, and gains, as follows:

- high grade oxide ore contributed about 25% of the tonnage in the 2023 mine claim, with an indicated metal loss of 3.7%
- high grade mixed ore contributed about 19% of the tonnage, with an indicated metal loss of 11.5%
- conversely, sulphide ore also contributed about 19% of the tonnage, but with an apparent metal gain of around 6%
- the mineralised waste contributed about 16% of the tonnage in the 2023 mine claim, with an average metal loss of about 5.4% indicated

This apparent variability points to a degree of post-blast ore/waste mixing during the digging process. Despite the variability apparent in the 2023 dispatch records, there is an overall 6% metal loss indicated for the relatively small proportion of high metal risk ore codes. The substantially larger low and medium risk proportions have an average 4% metal loss. Given these indications and the changing mining circumstances beyond 2023, the following adjustments were adopted for the matrix assisted calculation of unplanned metal losses in Table 16-1:

- high risk metal loss adjustment = 6%
- medium risk metal loss adjustment = 1.5%
- low risk metal loss adjustment = 0.5%

The adjustments for the medium and low risk metal losses were calibrated by means of a risk and tonnage weighted preliminary LOM schedule. Loss allowances due to blast movement, and reflecting operational improvements going forward, are captured in the adjustments applied by cutback (refer to Table 16-1).

[7] In this reconciliation process, the ex-pit FMS records are not reconciled against the process plant claim. The reason for this is that the former includes ore direct fed to the crushers and hauled to stockpiles, whereas the latter includes ore direct fed to and reclaimed from stockpiles to the crushers.

Table 16-2 2023 reconciliation of ore marked out vs ore dispatched

Ore Code	Prop'n of Total	Ore Types	POST BLAST MARKOUT		TO CRUSHERS & S/PILES		TO OTHER (DUMP)		Combined
			Metal Tonnes	Prop'n (%)	Metal Tonnes	Loss (%)	Metal Tonnes	Loss (%)	Metal Loss (%)
		OXIDE							
12	7%	LG_OX	16,140	7%	17,116	-6.0%	9	0.1%	-6.0%
14	25%	HG_OX	60,773	25%	58,626	3.5%	101	0.2%	3.7%
22	1%	OX_LG_HGAC	2,577	1%	2,172	15.7%	1	0.1%	15.8%
24	6%	OX_HG_HGAC	13,839	6%	11,396	17.7%	43	0.4%	18.0%
	38%		**93,329**	**38%**	**89,310**	**4.3%**	**155**	**0.2%**	**4.5%**
		MIXED							
52	5%	LG_MIXED	12,690	5%	11,483	9.5%	17	0.1%	9.7%
54	19%	HG_MIX	46,107	19%	41,016	11.0%	199	0.5%	11.5%
	24%		**58,796**	**24%**	**52,499**	10.7%	**216**	**0.4%**	**11.1%**
		SULPHIDE							
42	4%	LG_SUL	9,307	4%	9,380	-0.8%	46	0.5%	-0.3%
44	19%	HG_SUL	47,740	19%	50,922	-6.7%	189	0.4%	-6.3%
	23%		**57,047**	**23%**	**60,302**	**-5.7%**	**234**	**0.4%**	**-5.3%**
		OXIDE MW							
11	11%	OX_MW	26,492	11%	25,342	4.3%	111	0.4%	4.8%
	11%		**26,492**	**11%**	**25,342**	**4.3%**	**111**	**0.4%**	**4.8%**
		SUL MW							
41	5%	SUL_MW	12,173	5%	11,583	4.8%	131	1.1%	6.0%
	5%		**12,173**	**5%**	**11,583**	**4.8%**	**131**	**1.1%**	**6.0%**
		ALL ORE TYPES							
			247,837	**100%**	**239,036**	**3.6%**	**847**	**0.4%**	**3.9%**
		ALL - BASED ON METAL RISK ASSOCIATED WITH 2023 TRACKING RECORDS							
LOW	75%	LOW	184,969	75%	178,947	3.3%	612	0.3%	**3.6%**
MEDIUM	19%	MEDIUM	48,037	19%	46,108	4.0%	183	0.4%	**4.4%**
HIGH	6%	HIGH	14,831	6%	13,981	5.7%	52	0.4%	**6.1%**
	100%		**247,837**	**100%**	**239,036**	**3.6%**	**847**	**0.4%**	**3.9%**

16.5 Ore codes and copper grade ranges

The current ore categorisation details are listed in Table 16-3. Continuing from the marginal cut-off grade calculations that were outlined in the 2015 and 2020 Technical Reports, the lower cut-off grade separating ore from waste remains at 0.20% TCu. Further discussion on the marginal cut-off grade is provided in Item 15.4.1.

Table 16-3 Ore codes and grade range determinants

Ore Code	Name	Description	Copper Grade Range	GAC	ASCu to TCu ratio	WEATHN
			OXIDE			
12	LG_OX	Low Grade Oxide; Lower GAC	>0.40% TCu and <0.65% TCu	<110 kg/t	>0.40	>=2
14	HG_OX	High Grade Oxide; Lower GAC	>0.65% TCu	<110 kg/t	>0.40	>=2
22	OX_LG_HGAC	Low Grade Oxide; Higher GAC	>0.40% TCu and <0.65% TCu	> 110 kg/t	>0.40	>=2
24	OX_HG_HGAC	High Grade Oxide; Higher GAC	>0.65% TCu	>110 kg/t	>0.40	>=2
			MIXED			
52	LG_MIXED	Low Grade Mixed Float	>0.40% TCu and <0.65% TCu		<0.40	<2
54	HG_MIX	High Grade Mixed Float	>0.65% TCu		<0.40	<2
			SULPHIDE			
42	LG_SUL	Low Grade Sulphide	>0.35% TCu and <0.50% TCu		<0.10	
44	HG_SUL	High Grade Sulphide	>0.50% TCu		<0.10	
			MIN WASTE			
11	OX_MW	Oxidised Mineralised Waste	>0.20% TCu and <0.40% TCu			
41	SUL_MW	Sulphide Mineralised Waste	>0.20% TCu and <0.35% TCu		<0.10	
			OTHER			
70	WST	Waste	<0.20% TCu			=4
71	REF	Refractory Ore				

16.6 Mining overview

16.6.1 Grade control

Conventional open pit grade control practices are in place at Kansanshi, incorporating RC drilling and sampling on a suitably designed drilling pattern and over multiple bench horizons.

16.6.2 Drilling and blasting

Production drill and blast operations at Kansanshi are carried out by FQM personnel. An explosives supply contractor manufactures bulk explosives in an on-site emulsion plant and delivers explosives under a “down-the-hole” contract.

The production drilling fleet (as at 1st January 2024) consists of:

- 8 x Sandvik Pantera DP1500i top hammer drill rigs (115 or 127 mm diameter holes)
- 16 x Sandvik D25K down-the-hole hammer drill rigs (165 mm diameter holes)
- 9 x Epiroc STR45-10LF top hammer drill rigs (115 mm to 140 mm diameter holes)

The drill fleet is planned to be increased over the next two years as the S3 Project comes on line. The increase will consist of additional new drill rigs as well as the decommissioning and replacement of a number of the older units.

Production drill and blast patterns are drilled on 5 m or 10 m high benches, as determined by the geological conditions. For production blasting on 5 m high benches, 115 mm diameter holes are used. Whereas for 10 m high benches, 127 mm to 165 mm diameter holes are currently used, increasing to 200 mm diameter with the arrival of new drills.

Trim blasts are designed according to the bench height and pit wall profile, and typically consist of two to three rows of 127 mm or 165 mm diameter holes and a number of rows of production holes. Wall control blasting requires the use of dedicated pre-split designs, involving packaged explosives, with patterns designed according to the rock mass properties and pit wall profile.

Blast movement (heave and throw) is minimised by designing initiation sequences relative to ore/waste blocks and then by tracking material movement using in-house developed machine learning technology. Electronic delay detonators provide flexibility to the engineers for production and wall control initiation design.

Design quality and consistency is managed through the use of a design approval document detailing the design parameters and expected blast outcomes. The document is reviewed by the drill and blast, mine planning, geology and geotechnical departments and is signed-off before being implemented. An in-field quality control process ensures that the design is executed to a high standard and that the metrics associated with each blast, i.e. hole depth, charge mass and stemming length, are recorded for each hole so that a comparison with the design can be made and blast designs then optimised.

Specific drilling and blasting parameters are detailed in Table 16-4 to Table 16-6.

Table 16-4 Kansanshi production drilling and blasting parameters

	Units	Production parameters				
Hole Diameter	mm	200	165	127	127	115
Bench Height	m	10	10	10	5	5
Burden	m	4.5	4	3.2	2.9	2.6
Spacing	m	5.2	4.6	3.7	3.3	3
Subdrill	m	1.5	1.5	1.2	1	1
Air Deck	m	0	0	0	0	0
Stemming	m	4.3	3.5	2.8	2.5	2.5
Explosives Type		Emulsion	Emulsion	Emulsion	Emulsion	Emulsion
Explosives Density	g/cm³	1.2	1.2	1.2	1.15	1.15
Volume/hole	m³	234	184	118	48	39
Charge Length	m	7.2	8	8.4	3.5	3.5
Charge Mass/hole	kg	271	205	128	51	42
Powder Factor	kg/m³	1.16	1.12	1.08	1.07	1.07

Table 16-5 Kansanshi wall control drilling and blasting parameters

	Units	Trim blast parameters	
Hole Diameter	mm	165	127
Bench Height	m	10	5
Burden	m	4	3.5
Spacing	m	3	3
Subdrill	m	0	0
Air Deck	m	3	0
Stemming	m	3	2.5
Explosives Type		Emulsion	Emulsion
Explosives Density	g/cm³	1.2	1.15
Volume/hole	m³	120	53
Charge Length	m	4	2.5
Charge Mass/hole	kg	103	36
Powder Factor	kg/m³	0.86	0.69

Table 16-6 Kansanshi pre-split drilling and blasting parameters

	Units	Presplit parameters	
Hole Diameter	mm	127	127
Bench Height	m	10	5
Burden	m	NA	NA
Spacing	m	1.5	1.5
Subdrill	m	0	0
Air Deck	m	3.6	2.4
Stemming	m	0	0
Explosives Type		25mm Package	
Explosives Density	g/cm³	1.18	1.18
Area/hole	m²	17	9
Charge Length	m	8.8	4.1
Charge Mass/hole	kg	5.1	2.4
Powder Factor	kg/m³	0.35	0.35

16.6.3 Primary crushing and conveying of ore

All ore hauled direct from the Kansanshi pits or reclaimed from stockpiles, is currently fed to primary crushers adjacent to a ROM pad at the existing plant site.

The primary crushing circuit for the S3 expanded plant will comprise semi-mobile, independent, gyratory crushers (ThyssenKrupp KB 63 x 89 or ThyssenKrupp KB 63 x 130) operating in open circuit. Each crusher will be positioned adjacent to the pit rim and remote from the S3 plant area, and crushed ore will be transported to the plant by an overland conveyor (Figure 16-7).

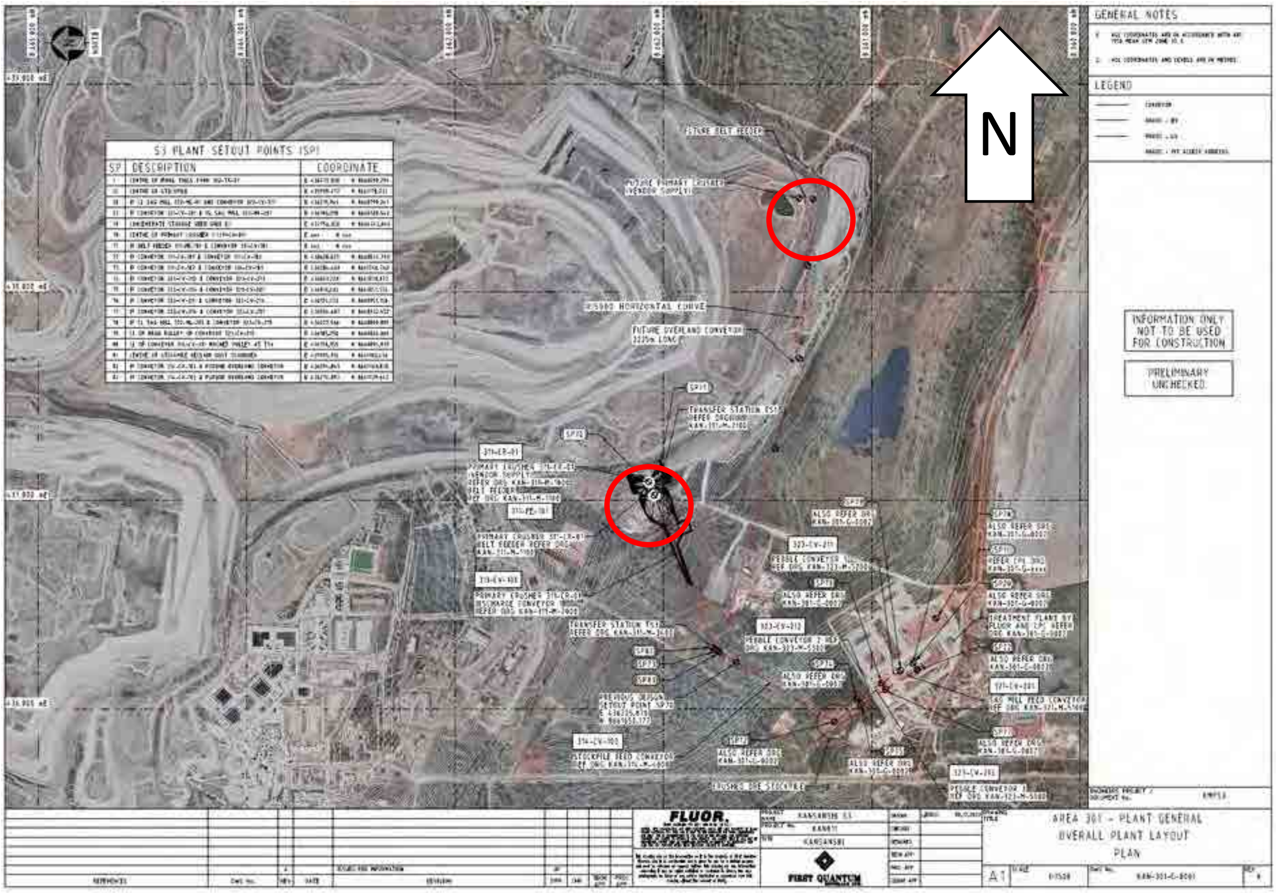
The ultimate pit designs currently show crushers located near-surface, and not subsequently relocated to lower positions in the deepening pits. This is a function of the current ramp designs and constraints which may be revised in the future. If feasible, and if future mining cutback phases and ramp designs can accommodate them, they may be relocated to sites within the deepening pits.

16.6.4 Trolley-assisted haulage

Towards the end of 2009 a mine electrification programme was implemented at Kansanshi to supplement the then existing diesel-electric trucks with an AC-drive fleet fitted with trolley assist (TA) pantographs. The benefits of trolley-assisted haulage include:

- reduction on diesel fuel consumption
- increase in up-ramp speed from 11 kmph for diesel haulage, to 23 kmph
- a 700 m length up-ramp trolley line improves cycle times by 7% per load on a 28 minute haul cycle
- a 700 m length up-ramp trolley line usage converts to a ~$3.50/hr truck engine overhaul saving

Figure 16-7 Primary crusher locations for the S3 expansion (source: Fluor)

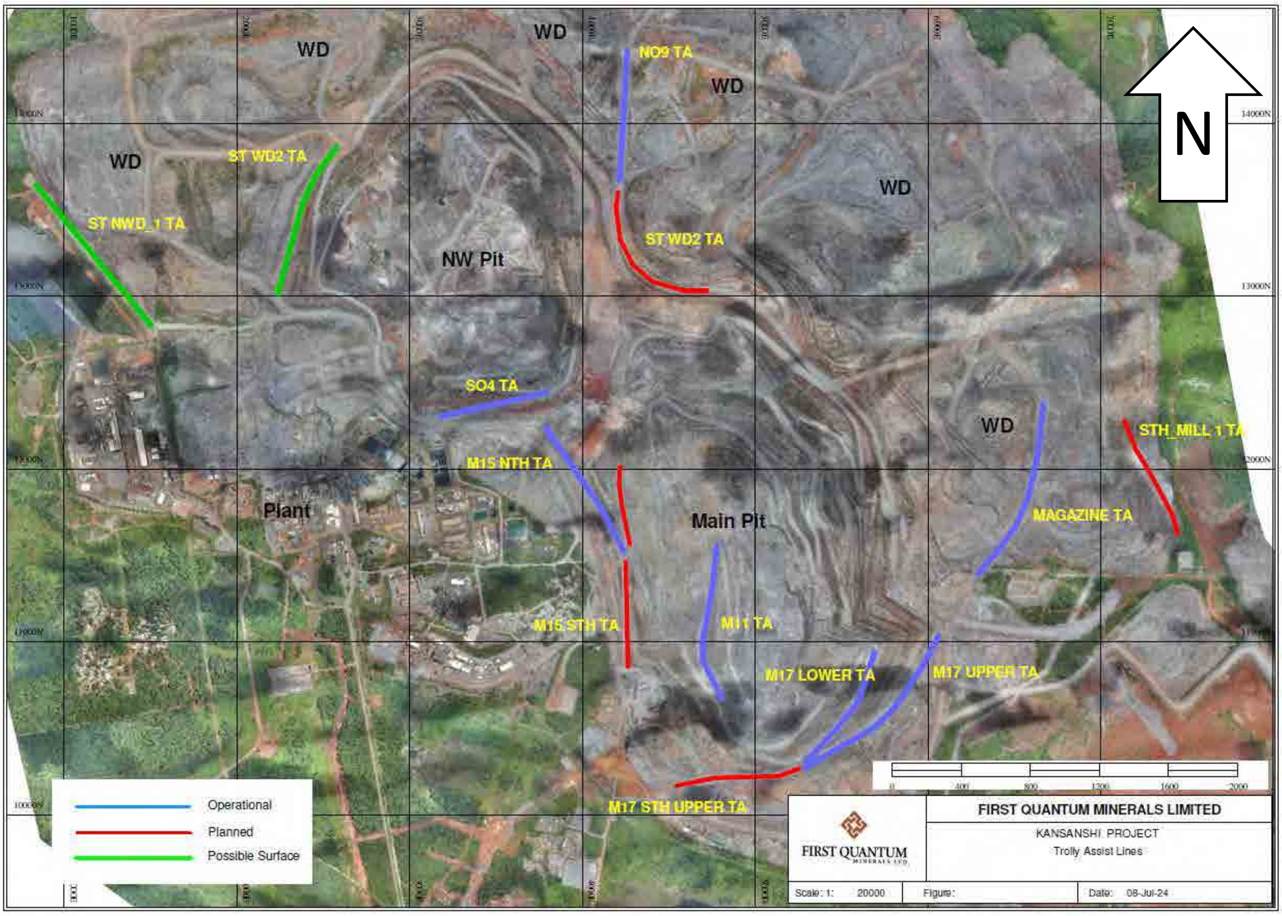


Currently operational TA ramps are shown in

Figure 16-8, along with several that are planned to be operational within the next five years, and others that could be constructed in the future. With reference to this figure:

- the M17 (South) Upper TA ramp is planned to be operational during Q1 2024
- the ST_M18_1 TA ramp is planned to be operational during Q2 2024
- the ST_MILL_1 TA ramp is planned to be operational during Q3 2024
- the N09 West TA ramp is planned to be operational during Q4 2024
- two possible surface TA ramps are shown as ST_NWD_1 and ST_W02, and each is subject to further design modifications
- a surface TA ring road around the South East Dome perimeter is also being considered

Figure 16-8 Trolley-assisted haulage routes (source: FQM)



16.6.5 Waste dumping

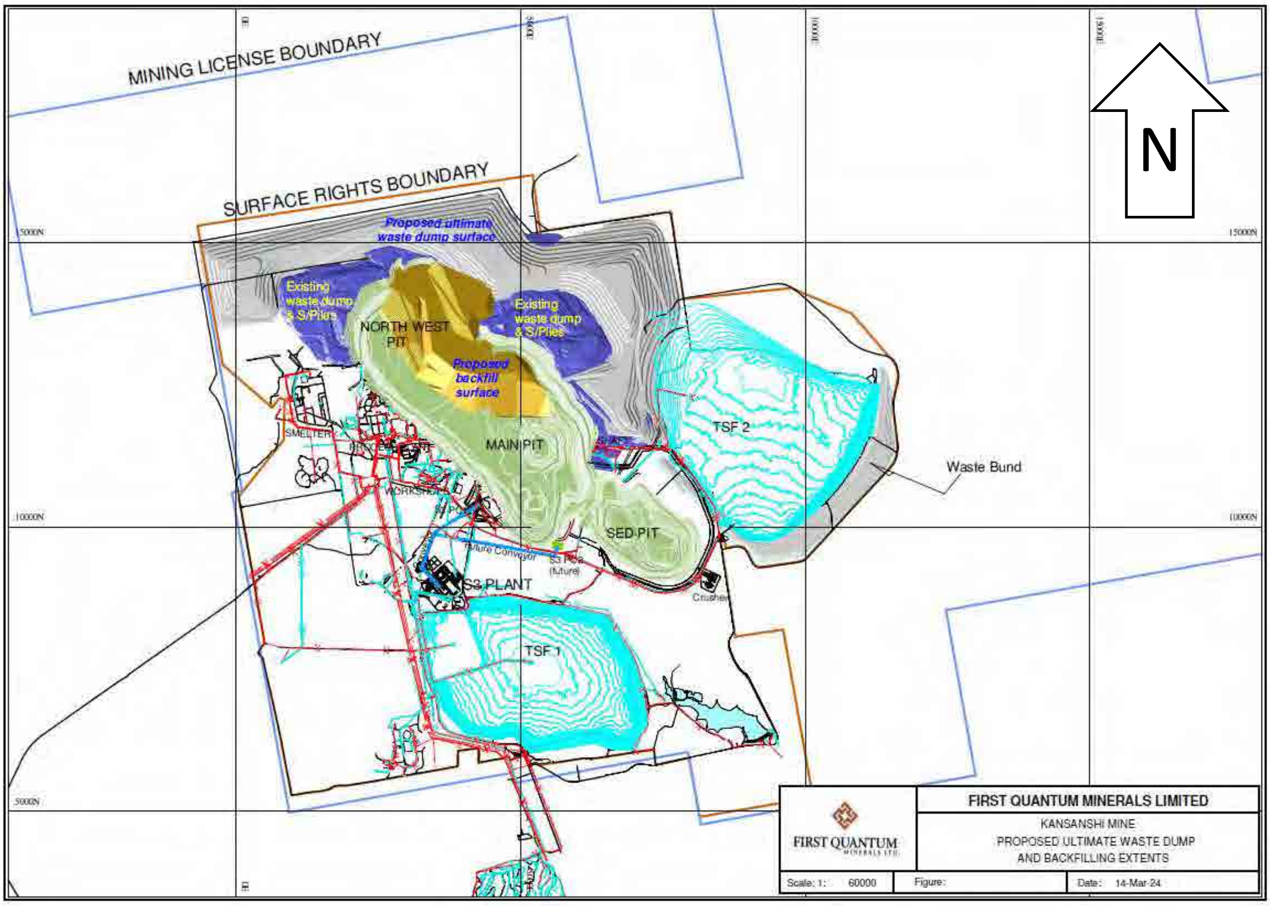
Figure 16-9 shows several existing waste dumps, as follows:

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

Inpit backfill capacity is available in the completed North West Pit, and on completion of the M10 cutback the backfill can be extended across to the north side of the pit. The backfilling site receives waste from mining of the adjacent Main Pit cutback phases, i.e. M15 and M16. A significant proportion of this waste is located between surface and 1,300 mRL. From 2035, the pit floor would be available to enable the remaining waste to be dumped inside of the pit.

Waste from the South East Dome Pit can be hauled initially to the TSF buttress where there is storage capacity for up to 30 million bcm. Current long-term stockpiles which are located on top of the waste dumps would be gradually reclaimed, thereby providing additional and nearby (i.e. short waste haul) surface dumping space after 2029.

Figure 16-9 Waste dump extensions and TSF waste buttress (source: FQM)



Over the term of the long term mine plan, the overall strip ratio will increase from an average of around 2.8 : 1, to an average of 4.8 : 1, in the period 2039 to 2042 during the last phases of mining Main Pit ore. A large proportion of the waste material will be overburden from near-to-surface cutbacks towards the ultimate limits (M21 and M28 in particular).

16.6.6 Ore stockpiles

Figure 16-3 shows the location of numerous surface ore stockpiles. Table 16-7, Table 16-8 and Table 16-9 show the current inventories for these stockpiles.

The production schedule (and Mineral Reserve statement in Table 15-12) include the reclaim and processing of ore from these stockpiles, a number of which are located on the waste dump located to the north of Main Pit. After reclamation, the space that was occupied by these stockpiles will be used for waste dumping.

Table 16-7 Oxide ore stockpiles, December 2023

Stockpile	MTonnes	TCu(%)	AsCu(%)
H_GAC_1	0.3	1.37	0.82
H_GAC_2	0.0	0.00	0.00
HG_OX_5	0.1	1.22	0.65
LG_HGAC_1	0.4	0.50	0.28
LG_OX_4	1.4	0.49	0.21
LG_OX_5	0.2	0.52	0.22
MWD_10	0.2	0.28	0.08
MWD_11	42.4	0.27	0.09
MWD_8	9.1	0.34	0.14
Total	**54.1**	**0.30**	**0.11**

Table 16-8 Mixed ore stockpiles, December 2023

Stockpile	MTonnes	TCu(%)	AsCu(%)
F8	0.1	1.02	0.27
HG_RED_6	0.3	0.59	0.17
LG_RED_1	15.2	0.62	0.20
LG_RED_4	16.4	0.58	0.20
LG_RED_5	11.4	0.52	0.15
LG_RED_6	2.3	0.47	0.11
O_GAC_3	0.0	0.00	0.00
Total	**45.7**	**0.57**	**0.18**

Table 16-9 Sulphide ore stockpiles, December 2023

Stockpile	MTonnes	TCu(%)	AsCu(%)
F9	0.1	0.75	0.01
F10	1.0	0.42	0.01
LG_SUL_3	3.4	0.48	0.01
LG_SUL_6	17.7	0.48	0.01
LG_SUL_7	12.0	0.39	0.01
LG_SUL_8	2.4	0.44	0.02
MG_SUL_1	0.1	0.62	0.02
MWD_12	20.6	0.27	0.01
MWD_9	12.4	0.27	0.02
Total	**69.7**	**0.36**	**0.01**

16.7 Mine planning considerations

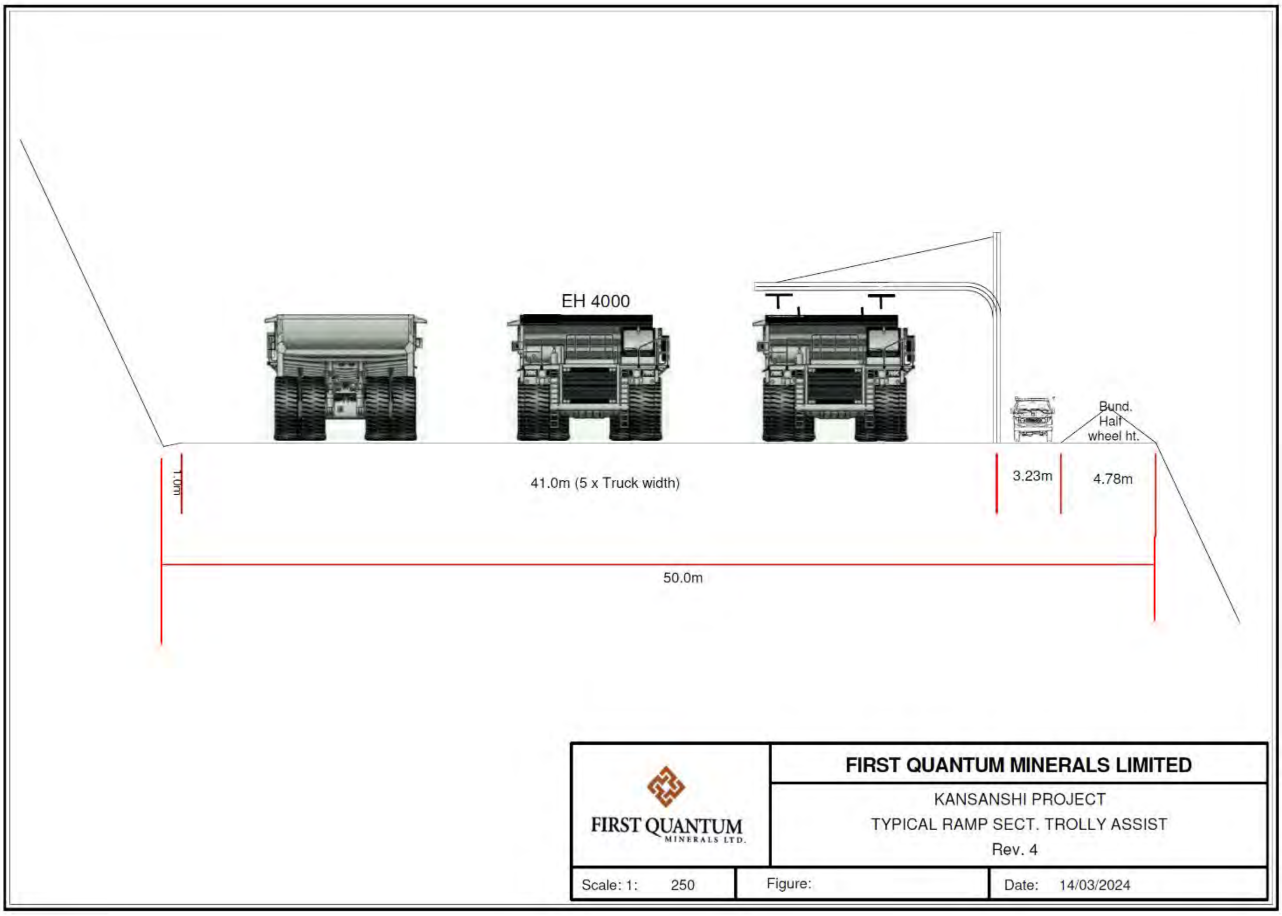
The following information relates to the detail that needed to be considered for designing surface layouts and practical mining pits around the optimal pit shell outlines. Other technical aspects are described which relate to the layout of the mining site.

16.7.1 Mine design parameters

Basic mine design parameters are described in Item 15. For more detailed planning, the following parameters were adopted in designing the pit layouts to suit trolley assisted truck haulage:

- haul road minimum width = 5 x EH4000 truck width (to allow up-haulage off the trolley line) = 41 m
- total haul road width inclusive of catenary pole, bund and side drain = 50 m (Figure 16-10)
- maximum gradient = 1 : 10

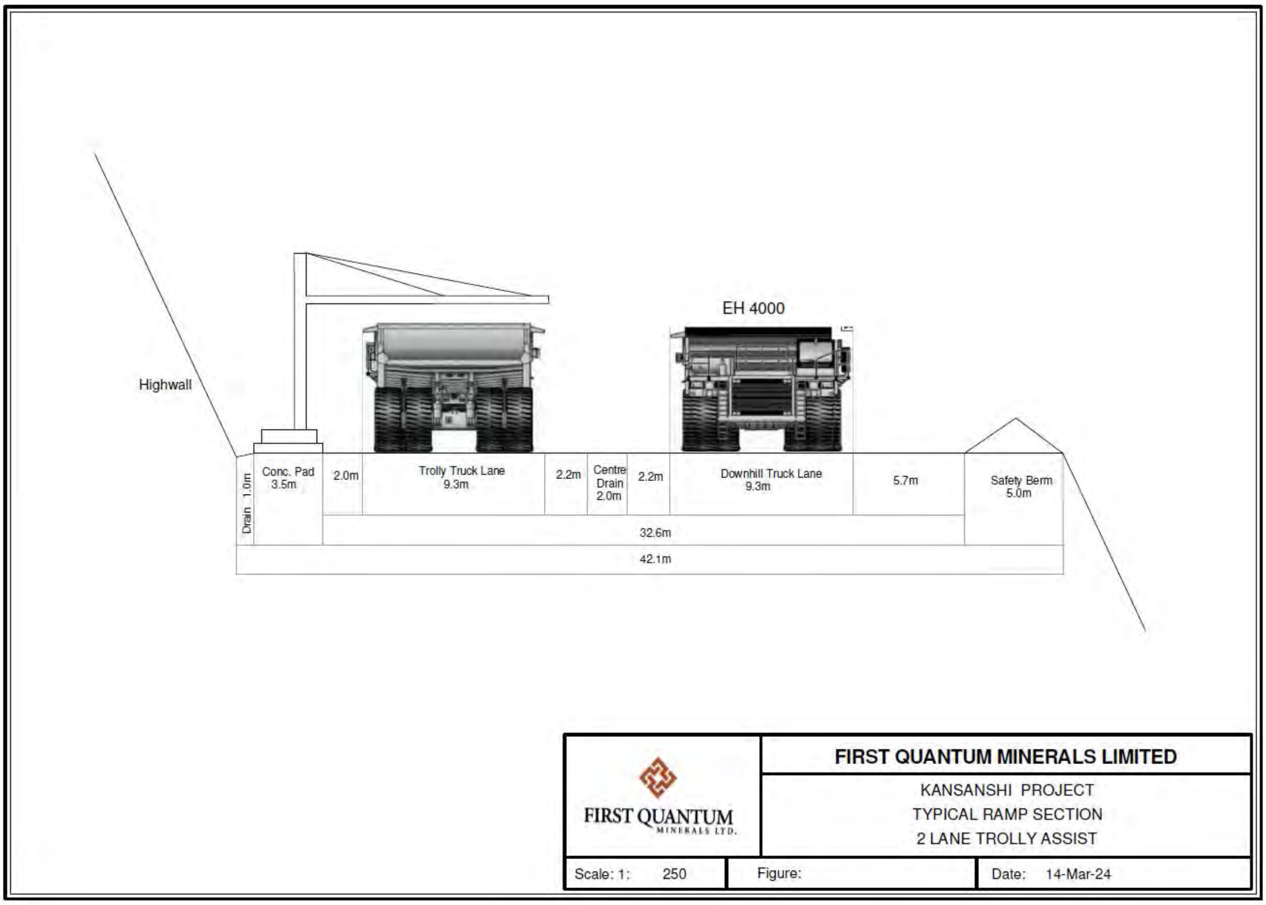
Figure 16-10 Schematic cross section across a three-lane trolley-assisted haulage ramp



Wherever there is only sufficient pit cutback width for a two-lane ramp, the specifications to accommodate trolley assisted truck haulage are as follows:

- haul road minimum width to suit EH4000 trucks = 32.6 m
- total haul road width inclusive of catenary pole, bund and drains = 42.1 m (Figure 16-11)
- maximum gradient = 1 : 10

Figure 16-11 Schematic cross section across a two-lane trolley-assisted haulage ramp



16.7.2 Mine geotechnical engineering

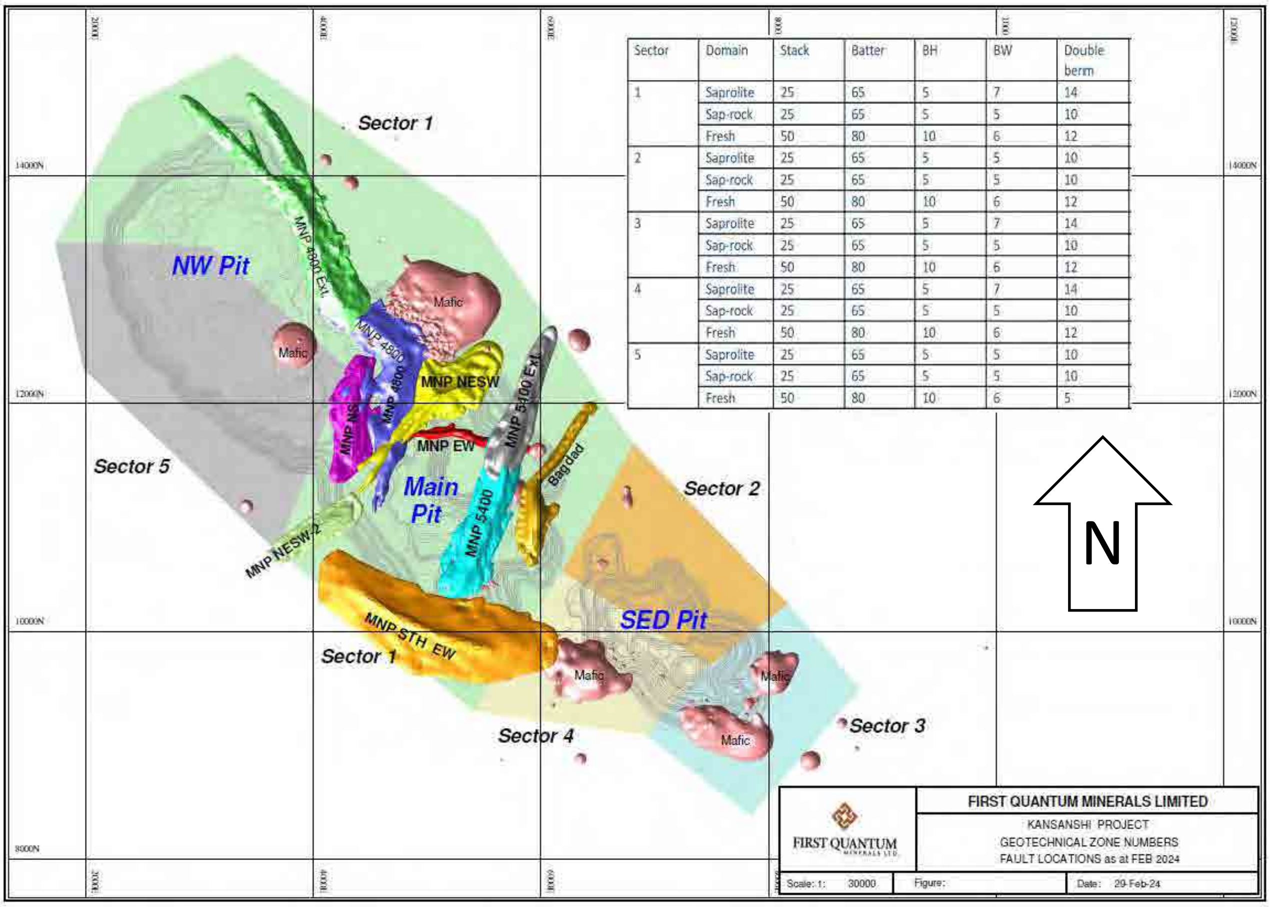
Geotechnical design sectors and pit slope design criteria were reviewed and updated in late 2019 for use in the 2020 Technical Report mine planning. In recognition of the then prevailing geological model and from confirmatory geotechnical drilling, the applicable design criteria was specified for ultimate pit slopes such that basal domain boundaries were defined by the geologically modelled weathering surfaces, rather than as conservative horizontal levels as perceived from then current pit exposures.

The geotechnical design criteria have since been further updated in accordance with the development of an improved understanding of fault zones and weathering horizons as captured in the latest geological block model (refer to Item 14). Figure 16-12 shows the revised interpretation of fault zones and pit slope design sectors, and also lists the latest recommended pit slope design criteria, as at December 2023.

The most notable changes from the 2020 Technical Report parameters are:

- 7 m width berms in saprolite are recommended to mitigate the risk of fault zones associated with deep weathering (up to 50 m)
- steepening of fresh rock slope angles is recommended where no faulting is present; actual performance of fresh rock slopes supports this change
- inclusion of double width geotechnical berms to arrest potential overspill from bench face instability

Figure 16-12 Kansanshi pit slope design sectors, 2023 (source: FQM)



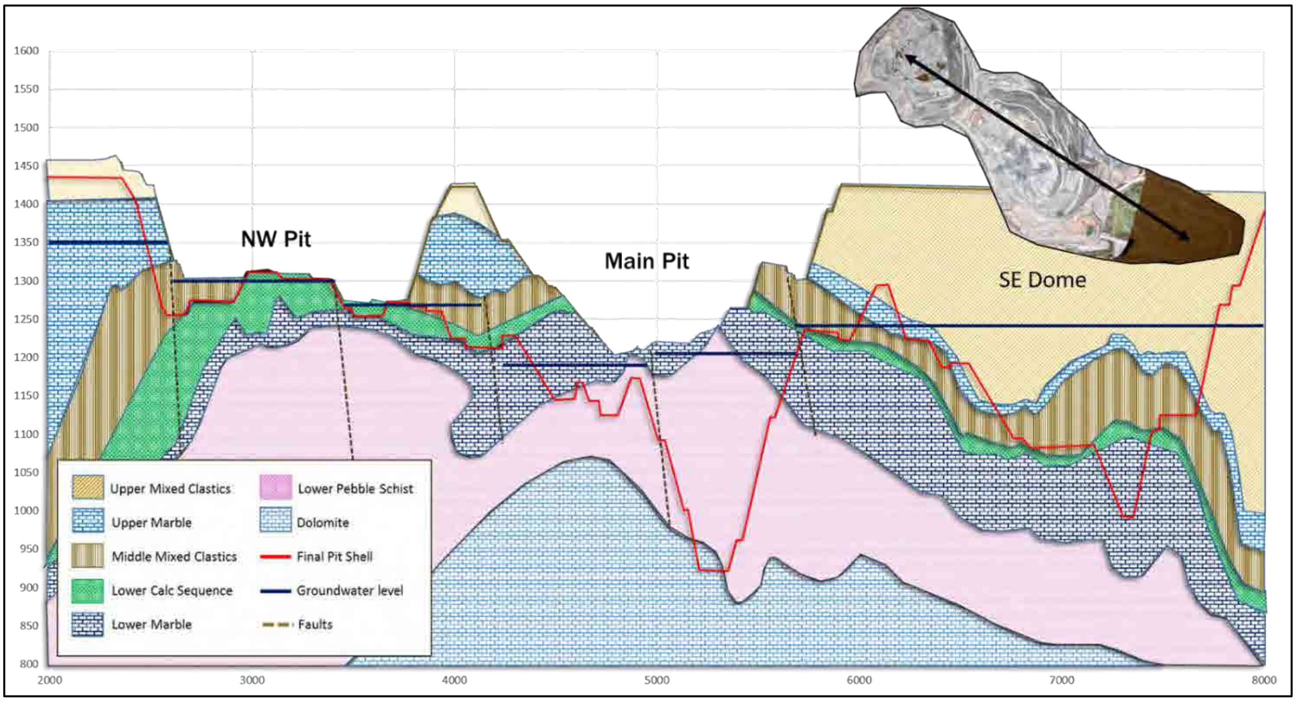
16.7.3 Hydrogeology and mine dewatering

The hydrogeological setting for the Kansanshi mining operations can be summarised as follows, and as shown in Figure 16-13:

- The regional groundwater flow direction is considered to be from the NNE to SSW.
- The rock types encountered at Kansanshi generally have low primary permeability (the rock fabric itself inhibits flow). The main flow is through secondary permeability, through mainly veins and fractures in the clastic rocks, and also through faults and associated cavities in the marble rocks.
- The Lower Marble unit and the Lower Calcerous Sequence are considered to be the main aquifers at Kansanshi. The groundwater can be stored in cavities and transported through highly weathered zones associated with structures.
- The groundwater system at Kansanshi can be divided into a series of discrete hydrogeological blocks that have similar groundwater levels and are interconnected.
- The NNE-SSW trending structures likely act as (partial) hydraulic boundaries to these blocks.
- The water gradient will be “stair-stepped” across the successive near vertical NNE-SSW trending structures with differences in hydraulic heads across these blocks.
- Virtually all water that enters the Kansanshi dewatering system is derived from ongoing recharge from rainfall and local runoff within the area surrounding the site.
- Regional geological structures and faults are considered to be significant conduits for the transportation of the groundwater recharge into the open pits.
- In addition, the groundwater in Main Pit is possibly recharged by surface water from adjacent dambo systems to the north.

- A secondary aquifer with a shallow water table is present and is associated with the saprolite rock type. Porosity and permeability in this unit are considered to be very low. This unit is significant in terms of pore pressures in open pit slopes.

Figure 16-13 Conceptual hydrogeological model of the Kansanshi open pits (source: FQM)



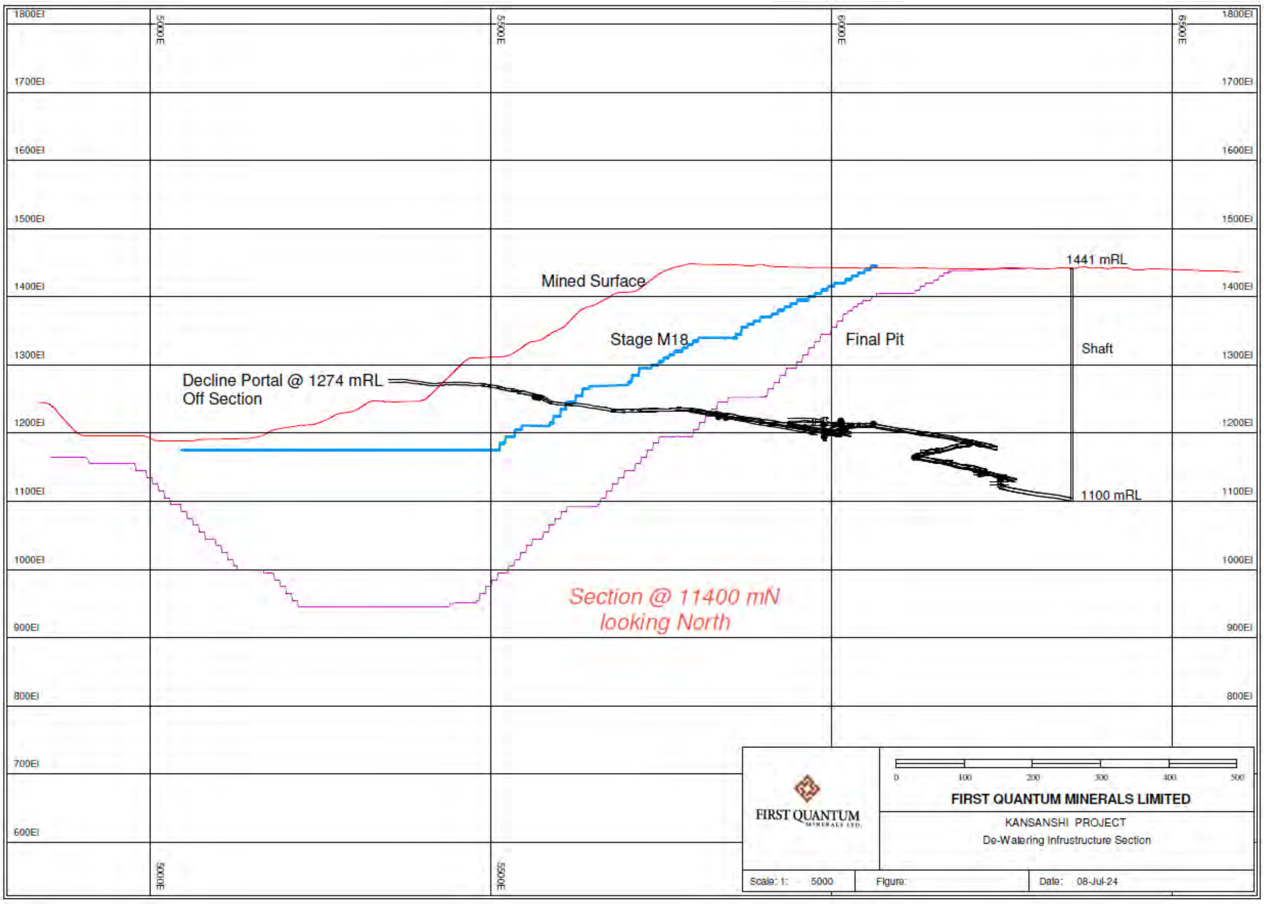
Dewatering of the Main Pit was previously carried out by means of an underground decline and a 4.1 m diameter, vertically raise-bored shaft located to the east of the M6 cutback phase. Water was pumped from the 200 m deep shaft, at rates up to 3,000 m³/hr, and piped to a storage dam to supplement the process water demand.

When groundwater levels had to be lowered to suit the planned deepening of pit levels (>200 m from surface), a new portal to the decline was established, where it was exposed on the M12 cutback face. A decline extension, branching off from the existing development, and extending to a level below the contact of the Lower Marble unit with the LOM pit shell is currently in the process of being developed. The new shaft will be raise-bored from this level (1100 RL, 340 m below topographic surface) and equipped with a submersible pump, to assist with the continued dewatering of the Main Pit and the South East Dome Pit. Figure 16-14 shows the current and planned pit dewatering development.

Currently, the Main Pit is dewatered by a series of in-pit sumps and pumps with the water exiting the pit via HDPE piping rising up the pit walls. In addition to this there are also a series of in-pit dewatering bores which are located in the M12 cutback. These pump water at a rate of 1,000 m³/hr and also exit the pit via piping infrastructure that rises up the high wall.

Dewatering of the North West Pit is carried out by means of collection and pumping from in-pit sumps.

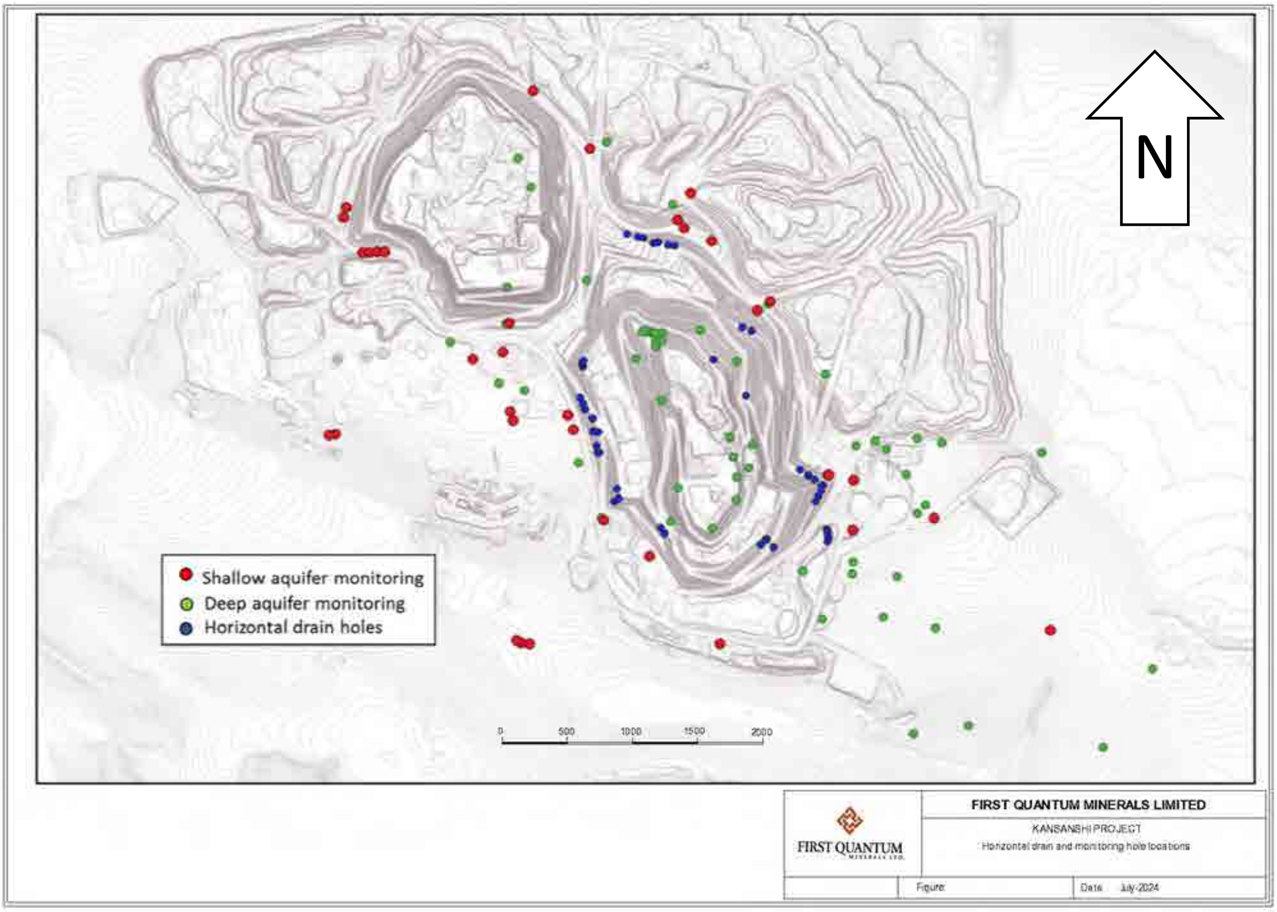
Figure 16-14 Cross section through existing and new decline extensions, and the planned new dewatering shaft (source: FQM)



As part of the slope depressurisation programme, a number of horizontal drain holes have been drilled at the Main Pit. These holes were drilled into the open face of the wall to reduce pore pressures and to depressurise areas behind pit walls. They were drilled at different orientations to cover an optimal sub-surface area.

In addition to the current network of approximately 50 multi-level vibrating wireline piezometers (VWP's) installed around the Main Pit, holes were drilled behind pit walls and then equipped to measure pore pressures. These installations will allow the monitoring of pore pressure changes over time as mining progresses and therefore the efficiency of the depressurisation holes. Data from the horizontal drilling programme will be used for future slope design and potential slope optimisation.

Figure 16-15 Horizontal drain and monitoring hole locations in and around the open pits (source: FQM)



16.7.4 ARD management

According to the Company's Ground Control Management Plan (KMP, 2015), a geochemical waste characterisation study has been carried out on selected waste rock samples from the Kansanshi mine. The findings are that laterite, saprolite and saprock can be classified as non-acid forming (NAF). The marbles can be classified as NAF, despite containing traces of sulphides in a groundmass of calcite. The knotted schists and phyllites located away from the centre of the orebody can be classified as potentially acid forming, along with phyllites that are proximal to the centre of the orebody. According to the 2002 ESIA for KMP (Blandford, 2002), the presence of carbonate lithologies should extend the acid generating lag phase considerably.

Despite the presence of sulphide minerals that exist in the 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, due to the existence of acid neutralising carbonates contained within the waste material (KMP, 2015).

Geochemical testing in 2023 highlighted the variability in terms of ARD potential of the different stratigraphic and lithologic units that will be disturbed by mining of the South East Dome deposit. Stratigraphic units that may pose a potential ARD risk are the Upper Mixed Clastics and Middle Mixed Clastics, especially carbonaceous lithologies within these stratigraphic units. In contrast, the marble and pebble schist units were generally net neutralising. When considered as a whole, there is sufficient neutralising potential within the full stratigraphic column to mitigate the acidity generated by the Upper Mixed Clastics and Middle Mixed Clastics within the sampled material. KMPi is developing a testing and characterisation programme to ensure appropriate management of potentially acid generating material.

16.8 Mining and processing schedules

With the completion of the detailed ultimate pit designs, detailed life-of-mine (LOM) production scheduling was completed using MineSched software.

16.8.1 LOM schedule

Table 16-10 to Table 16-13 summarise the physicals for the Kansanshi life of mine (LOM) schedule.

Table 16-10 Kansanshi LOM schedule, ore and waste mining (after planned and unplanned mining losses)

Year	Mined Ore								Total	Mined	Strip ratio
	Oxide (Mt)	Mixed (Mt)	Sulphide (Mt)	Main Pit (Mt)	NW Pit (Mt)	SED Pit (Mt)	Total (Mt)	Grade (%TCu)	Mined (Mt)	Waste (Mt)	
2024	8.7	5.6	13.4	27.3		0.4	27.7	0.74	103.5	75.8	2.7
2025	17.4	7.7	20.9	31.6	4.3	10.2	46.1	0.71	153.9	107.9	2.3
2026	20.8	5.6	19.0	27.5	1.4	16.5	45.4	0.55	161.7	116.3	2.6
2027	14.4	4.9	24.9	19.7	0.9	23.6	44.2	0.59	178.6	134.4	3.0
2028	19.7	3.5	26.7	18.3	5.6	26.1	49.9	0.55	188.9	139.0	2.8
2029	22.6	4.8	25.9	28.8	10.6	14.0	53.4	0.56	206.7	153.4	2.9
2030	15.3	7.0	39.2	37.3	8.1	16.2	61.6	0.57	209.2	147.6	2.4
2031	11.9	6.0	36.1	30.0	8.5	15.5	53.9	0.61	211.8	157.8	2.9
2032	8.8	5.9	37.6	37.9	2.6	11.8	52.3	0.60	206.4	154.1	2.9
2033	7.0	5.9	45.0	44.2		13.8	57.9	0.58	209.4	151.4	2.6
2034	7.0	4.8	48.6	45.1		15.2	60.3	0.52	222.1	161.8	2.7
2035	9.1	5.4	44.5	40.6		18.4	59.0	0.53	217.8	158.7	2.7
2036	10.2	4.4	37.5	38.5	0.1	13.6	52.1	0.55	203.7	151.6	2.9
2037	9.9	3.7	31.6	36.7	0.6	7.8	45.1	0.55	203.2	158.1	3.5
2038	10.6	2.4	35.0	29.9	8.0	10.1	48.0	0.55	197.3	149.4	3.1
2039	9.1	2.4	22.1	19.8	13.4	0.4	33.5	0.57	161.6	128.1	3.8
2040	2.3	0.8	28.9	17.0	15.0		32.0	0.51	176.9	144.9	4.5
2041	1.5	0.3	15.0	16.8			16.8	0.52	130.1	113.3	6.8
2042	0.6	0.1	23.0	23.7			23.7	0.58	122.5	98.8	4.2
2043	1.9	0.2	26.5	28.7			28.7	0.47	91.8	63.1	2.2
2044	1.7	0.2	19.6	21.6			21.6	0.44	59.3	37.7	1.7
2045			11.4	11.4			11.4	0.35	20.6	9.2	0.8
2046			10.6	10.6			10.6	0.31	16.5	5.9	0.6
2047											
2048											
2049											
2050											
TOTAL	**210.6**	**81.5**	**643.1**	**643.0**	**79.0**	**213.2**	**935.2**	**0.56**	**3,653.6**	**2,718.3**	**2.9**

Relative to the 2020 Technical Report LOM schedule, and bearing in mind the mining depletion since that time, the total mined ore tonnage has increased by 11%, whilst the insitu copper metal is around 3% less than that reported in 2020. The total pit tonnage has reduced by 12% and the overall strip ratio is now 2.9 as opposed to 3.7 in 2020. Another two years of mining is now scheduled (Table 16-10). Table 16-11 lists the annual plant feed schedule, by ore feed type, and now extending to 2049. Relative to the 2020 Technical Report, and whilst the oxide and mixed feed tonnages have reduced as a function of mining depletion, the sulphide feed inventory has increased by 23%.

Table 16-12 lists the annual plant feed schedule in terms of insitu and recovered copper and gold. The overall projected average copper recovery is about 3% lower than that reported in the 2020 Technical Report, partially attributable to revised mining loss factors.

Table 16-13 lists the annual stockpile movements, comprising reclaim from the existing stockpiles, together with building and reclaiming from new stockpile movements. To note are:

- oxide stockpile on-movements in excess of 80 Mt for the period to 2030, with gradual reclaim until the final year of mining in 2044, to be followed by over 100 Mt of reclaim in the final five years of the LOM time frame
- progressive reclaim of mixed ore stockpiles (existing and new) over the course of mining to 2044
- sulphide stockpile off-movements in excess of 80 Mt for the period to 2030, with generally progressive reclaim for each year thereafter

In addition to the material movements listed in Table 16-10, there are waste volumes required to be moved during the period 2024 to 2026. Approximately 12.7 Mbcm (or 32.2 Mt) will need to be relocated from existing waste dumps to accommodate Main Pit cutbacks.

Table 16-11 Kansanshi LOM schedule, plant feed by circuit (after planned and unplanned mining losses)

Year	Direct Feed	Reclaim Feed	Oxide Feed			Mixed Feed			Sulphide Feed			Total Feed		
	(Mt)	(Mt)	(Mt)	(%TCu)	(g/t Au)	(Mt)	(%TCu)	(g/t Au)	(Mt)	(%TCu)	(g/t Au)	(Mt)	(%TCu)	(g/t Au)
2024	13.0	15.7	7.3	0.63	0.14	7.8	0.81	0.22	13.6	0.64	0.15	28.7	0.68	0.16
2025	21.8	20.1	7.3	0.65	0.15	7.8	1.06	0.26	27.4	0.49	0.12	42.5	0.62	0.15
2026	26.5	26.5	7.2	0.59	0.14	7.8	0.75	0.13	38.5	0.41	0.09	53.5	0.49	0.11
2027	33.8	19.5	7.2	0.56	0.16	12.4	0.59	0.11	33.7	0.54	0.10	53.3	0.56	0.11
2028	33.9	19.7	7.3	0.60	0.10	7.8	0.55	0.10	38.6	0.55	0.11	53.7	0.55	0.10
2029	36.5	17.1	7.3	0.75	0.12	7.8	0.62	0.10	38.5	0.50	0.10	53.6	0.55	0.10
2030	48.3	5.3	7.3	0.69	0.11	7.8	0.90	0.15	38.5	0.54	0.10	53.6	0.61	0.11
2031	47.8	5.7	7.3	0.75	0.11	7.8	0.76	0.13	38.5	0.56	0.10	53.6	0.61	0.11
2032	49.3	6.9	7.3	0.61	0.12	7.8	0.70	0.12	41.0	0.56	0.11	56.1	0.58	0.11
2033	52.1	3.9	7.3	0.61	0.11	7.8	0.75	0.14	40.9	0.56	0.11	56.0	0.59	0.12
2034	50.5	5.4	7.3	0.45	0.08	7.8	0.63	0.10	40.9	0.55	0.10	56.0	0.55	0.10
2035	51.4	4.6	7.3	0.45	0.08	7.8	0.65	0.11	40.9	0.55	0.10	56.0	0.55	0.10
2036	47.2	8.9	7.3	0.55	0.09	7.8	0.63	0.10	41.0	0.52	0.10	56.1	0.54	0.10
2037	40.1	15.8	7.3	0.46	0.10	7.8	0.60	0.10	40.9	0.51	0.10	56.0	0.51	0.10
2038	46.2	9.6	7.3	0.59	0.11	13.9	0.48	0.07	34.5	0.56	0.13	55.7	0.55	0.11
2039	33.5	22.2	7.3	0.70	0.11	13.9	0.37	0.08	34.5	0.47	0.10	55.7	0.47	0.10
2040	27.4	0.0							27.5	0.55	0.12	27.5	0.55	0.12
2041	15.5	11.9							27.4	0.44	0.10	27.4	0.44	0.10
2042	23.2	4.2							27.4	0.54	0.11	27.4	0.54	0.11
2043	27.0	0.4							27.4	0.47	0.10	27.4	0.47	0.10
2044	20.2	7.3							27.4	0.41	0.09	27.4	0.41	0.09
2045	11.4	15.9							27.4	0.31	0.09	27.4	0.31	0.09
2046	10.6	16.8							27.4	0.31	0.10	27.4	0.31	0.10
2047		27.4							27.4	0.33	0.07	27.4	0.33	0.07
2048		27.5							27.5	0.33	0.07	27.5	0.33	0.07
2049		19.5							18.1	0.33	0.07	18.1	0.33	0.07
2050														
TOTAL	**767.2**	**337.6**	**116.8**	**0.60**	**0.12**	**141.4**	**0.65**	**0.12**	**846.6**	**0.49**	**0.10**	**1,104.7**	**0.52**	**0.11**

Table 16-12 Kansanshi LOM schedule, plant feed by circuit (after planned and unplanned mining losses)

Year	Direct Feed	Reclaim Feed	Total Feed			Insitu metal		Rec'd metal		Overall recovery (%)	
	(Mt)	(Mt)	(Mt)	(%TCu)	(g/t Au)	(kt Cu)	(koz Au)	(kt Cu)	(koz Au)	(Cu)	(Au)
2024	13.0	15.7	28.7	0.68	0.16	195.9	151.0	157.6	49.2	80.5%	32.6%
2025	21.8	20.1	42.5	0.62	0.15	263.4	211.1	215.1	70.7	81.7%	33.5%
2026	26.5	26.5	53.5	0.49	0.11	260.0	183.2	213.7	62.4	82.2%	34.1%
2027	33.8	19.5	53.3	0.56	0.11	296.5	191.4	241.9	64.5	81.6%	33.7%
2028	33.9	19.7	53.7	0.55	0.10	297.4	178.5	248.9	62.4	83.7%	35.0%
2029	36.5	17.1	53.6	0.55	0.10	293.8	180.2	242.2	62.2	82.4%	34.5%
2030	48.3	5.3	53.6	0.61	0.11	327.6	192.5	270.8	66.5	82.7%	34.6%
2031	47.8	5.7	53.6	0.61	0.11	329.2	182.2	273.8	62.9	83.2%	34.5%
2032	49.3	6.9	56.1	0.58	0.11	327.9	207.1	274.6	72.1	83.7%	34.8%
2033	52.1	3.9	56.0	0.59	0.12	331.3	211.0	276.8	73.7	83.5%	34.9%
2034	50.5	5.4	56.0	0.55	0.10	306.4	179.8	259.3	63.4	84.6%	35.3%
2035	51.4	4.6	56.0	0.55	0.10	309.1	184.1	261.4	64.8	84.6%	35.2%
2036	47.2	8.9	56.1	0.54	0.10	302.3	174.1	253.6	61.0	83.9%	35.0%
2037	40.1	15.8	56.0	0.51	0.10	287.6	185.4	240.5	65.1	83.6%	35.1%
2038	46.2	9.6	55.7	0.55	0.11	304.2	199.0	249.6	69.2	82.0%	34.8%
2039	33.5	22.2	55.7	0.47	0.10	264.2	172.2	211.3	59.1	80.0%	34.3%
2040	27.4	0.0	27.5	0.55	0.12	150.5	102.1	135.6	38.3	90.1%	37.5%
2041	15.5	11.9	27.4	0.44	0.10	119.7	90.5	102.3	33.9	85.5%	37.5%
2042	23.2	4.2	27.4	0.54	0.11	149.1	99.9	133.5	37.5	89.5%	37.5%
2043	27.0	0.4	27.4	0.47	0.10	130.0	91.9	117.2	34.5	90.2%	37.5%
2044	20.2	7.3	27.4	0.41	0.09	111.8	79.2	96.0	29.7	85.8%	37.5%
2045	11.4	15.9	27.4	0.31	0.09	85.7	77.9	68.4	29.2	79.8%	37.5%
2046	10.6	16.8	27.4	0.31	0.10	84.5	87.8	65.8	32.9	77.9%	37.5%
2047		27.4	27.4	0.33	0.07	91.3	58.5	62.5	21.9	68.4%	37.5%
2048		27.5	27.5	0.33	0.07	91.6	58.6	62.6	22.0	68.4%	37.5%
2049		19.5	18.1	0.33	0.07	60.2	38.6	41.2	14.5	68.4%	37.5%
2050											
TOTAL	**767.2**	**337.6**	**1,104.7**	**0.52**	**0.11**	**5,771.1**	**3,767.7**	**4,776.3**	**1,323.7**	**82.8%**	**35.1%**

Table 16-13 Kansanshi LOM schedule, stockpile movements

Year	Oxide Feed			Mixed Feed			Sulphide Feed		
	On (Mt)	Off (Mt)	Balance (Mt)	On (Mt)	Off (Mt)	Balance (Mt)	On (Mt)	Off (Mt)	Balance (Mt)
2023			54.1			45.7			69.7
2024	7.4	6.0	55.5	1.0	3.2	43.5	6.3	6.5	69.6
2025	14.5	4.4	65.6	1.5	1.6	43.4	8.2	14.7	63.0
2026	14.5	0.9	79.2	0.2	2.4	41.3	3.7	23.2	43.5
2027	7.8	0.6	86.4	0.0	7.5	33.7	2.6	11.5	34.7
2028	12.8	0.3	98.8	1.7	6.0	29.4	1.6	13.5	22.8
2029	15.3	0.0	114.1	1.6	4.5	26.4	0.0	12.6	10.2
2030	8.6	0.6	122.2	0.9	1.7	25.7	3.7	3.0	10.9
2031	5.1	0.5	126.8	0.6	2.4	23.9	0.4	2.9	8.5
2032	2.6	1.1	128.3	0.5	2.4	22.0	0.0	3.4	5.0
2033	1.6	1.8	128.0	0.2	2.1	20.1	4.1	0.0	9.1
2034	2.1	2.4	127.7		3.0	17.1	7.7	0.0	16.8
2035	4.0	2.2	129.5		2.4	14.7	6.0	2.4	20.4
2036	4.9	2.0	132.4		3.4	11.3	0.0	3.5	17.0
2037	5.0	2.4	135.0		4.1	7.2	0.0	9.3	7.7
2038	1.2	2.3	133.9		7.2		0.5	0.0	8.2
2039	0.0	14.0	119.9				0.0	8.2	0.0
2040	2.1	0.0	122.0				2.4	0.0	2.4
2041	1.3	9.5	113.8					2.4	
2042	0.5	4.2	110.2						
2043	1.7	0.4	111.5						
2044	1.4	7.3	105.6						
2045		15.9	89.7						
2046		16.8	72.9						
2047		27.4	45.5						
2048		27.5	18.1						
2049		18.1							
2050									
TOTAL	**114.2**	**168.4**		**8.2**	**53.8**		**47.2**	**116.9**	

Features of the LOM mining and production schedule, as listed in these tables, are as follows:

1. The total material mined from all pits amounts to 3,653.6 Mt (1,420.8 Mbcm), of which 935.2 Mt is ore (including oxide, mixed and sulphide ore) and 2,718.3 Mt is waste (including refractory and Inferred Ore).
2. Mill feed rates in the first years of the schedule have been optimised based upon the feed availability and so vary slightly each year. From 2028 and for the remainder of the life of mine, the mill feed capacities are set at 7.3 Mtpa for oxide ore feed, 7.8 Mtpa for mixed ore feed, 13.5 Mtpa for sulphide ore in the existing S2 plant and up to an additional 27.5 Mtpa for the S3 plant. The exceptions to these set capacities are:
   – 2027, when the mixed feed plant processes 12.4 Mt to compensate for a lack of S2 plant feed
   – And again in 2038 and 2039 for the same reason, when the mixed feed plant processes 13.9 Mt in each year
3. Processing from the S3 plant will commence in H2 2025 (13.9 Mt processed); the plant will be fully operational in 2026. The feed rate will be 25 Mtpa for six years, increasing to a maximum of 27.5 Mtpa from 2032.
4. The LOM mining sequence is driven largely by opening up additional feed areas for the S3 sulphide expansion.

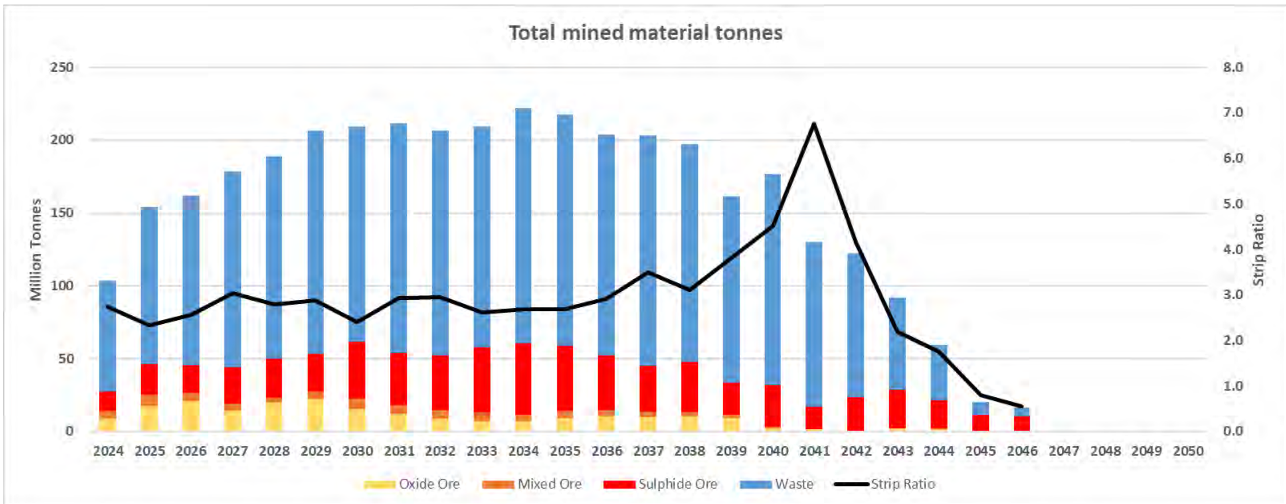
5. Annual BCM's ramp up to around 80 million over the period between 2024 and 2031, and remain at that level until 2038, thereafter trending downwards until the end of mining in 2046.
6. The majority of additional material being mined is characterised as near surface overburden waste material which can be bulk mined efficiently on wide terraced pushbacks.
7. In the mine sequencing, the higher strip ratio pushbacks are delayed towards the end of the schedule to limit the annual mining volumes.
8. Feed to the oxide, mixed and S2 circuits is complete in 2039 and the longer term Main and final North West Pit phase cutbacks commence from 2040, continuing to provide feed to the S3 circuit until 2046.
9. Main Pit is mined extensively throughout the life of the mine and has three main phases (namely m16, m21 and m28). It is completed three years ahead of processing completion in 2049.
10. The completion of North West Pit opens up space for waste backfill on the north wall.
11. Pre-strip mining of South East Dome began in 2024 and is scheduled for completion in 2039.

In terms of feed types to the separate processing facilities:

1. Sulphide feed to the S2 plant is currently high-grade ore from the Main Pit and the North West pit. Any low-grade sulphide encountered in mining is currently sent to long term surface stockpiles. When S3 commences production, there is additional capacity to process the low-grade sulphides directly from the pit and an opportunity to rehandle the long-term low grade and mineralised waste stockpiles. Between 2024 and 2029 a large proportion of the long-term sulphide stockpiles are rehandled into the ore feed.
2. The S3 ore feed is balanced between feed from Main Pit and from South East Dome, and depending on availability, some material is rehandled from the surface stockpiles. Sulphide ore feed from the remaining cutbacks of North West Pit is fed into the S2 sulphide mills along with ore from Main Pit and from stockpile reclaim.
3. Oxide and mixed ore feed continues to be provided from Main and North West with a small amount from the near surface material at South East Dome. However, after the S3 expansion, mining efforts focus on waste stripping and opening up faces to provide additional sulphide feed.

Figure 16-16 to Figure 16-23 depict the LOM schedule graphical results.

Figure 16-16 Kansanshi LOM schedule, annual material movement tonnes



Relative to a similar chart produced for the 2020 Technical Report (FQM, September, 2020), Figure 16-16 shows a peak of 220 Mt in 2034, as compared with 240 Mt in 2036 from the previous schedule. The strategy of deferring the high strip ratio cutbacks was also a feature of the 2020 LOM schedule.

Figure 16-17 shows the profile of annual component feed tonnes and the overall average annual feed grade into the combined circuits. Relative to the 2020 Technical Report, Figure 16-17 shows ten years of S3 processing after completion of S2 processing, as compared with seven years in the previous schedule.

Figure 16-17 Kansanshi LOM schedule, annual plant feed tonnes and grade into the combined circuits

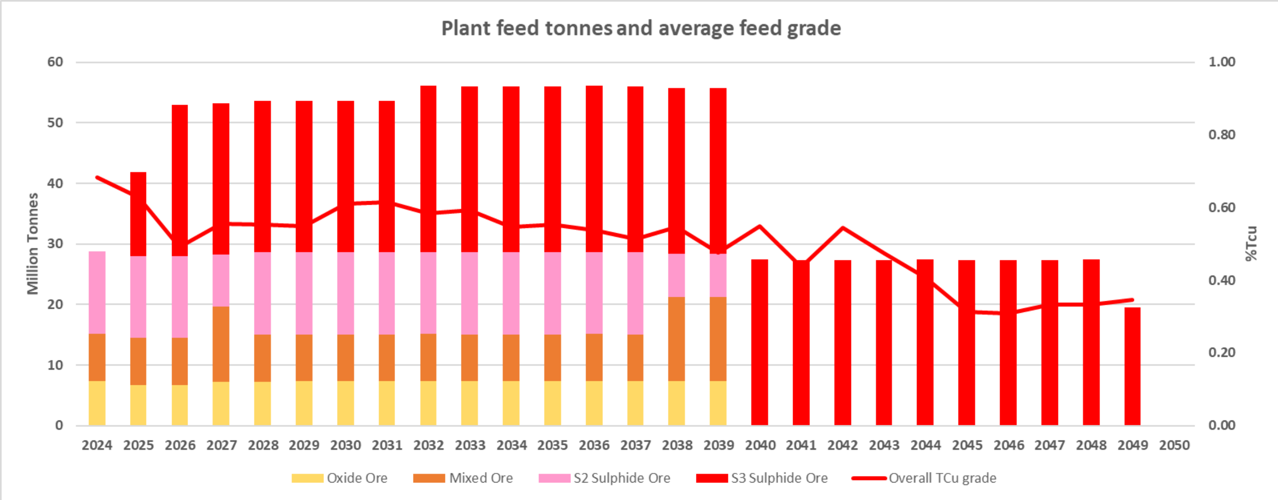


Figure 16-18 shows the annual oxide and mixed plant feed profiles. To note, is the mixed ore feed in those years where capacity is expanded to compensate for throughput constraints in the S2 plant. The corresponding S2 plant "troughs" are evident in Figure 16-19.

Figure 16-18 Kansanshi LOM schedule, annual oxide and mixed plant feed profiles

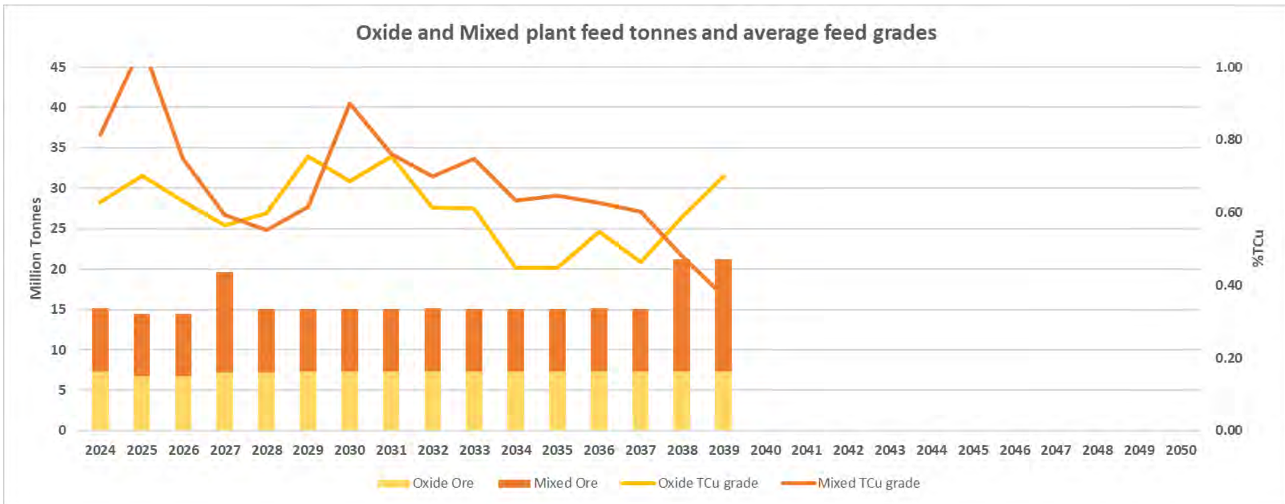


Figure 16-19 Kansanshi LOM schedule, annual sulphide plant (S2 and S3) feed profile

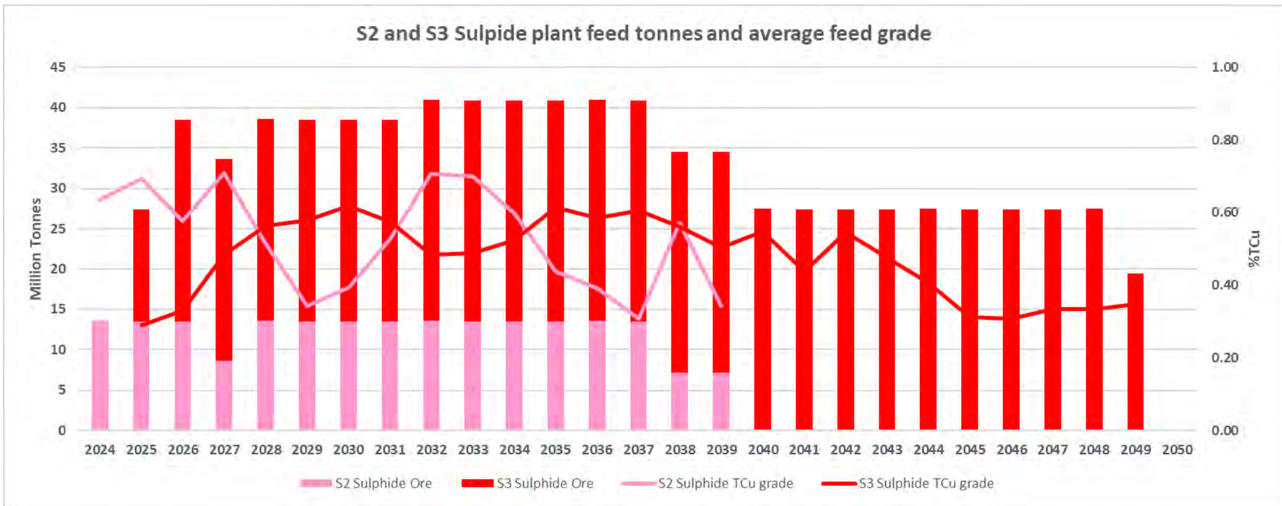


Figure 16-20 shows the annual copper metal recovered from each circuit and the corresponding process recovery rates. The combined average processing recovery of oxide leach and float feed is 73.2%.

Figure 16-20 Kansanshi LOM schedule, annual copper metal recovery and production

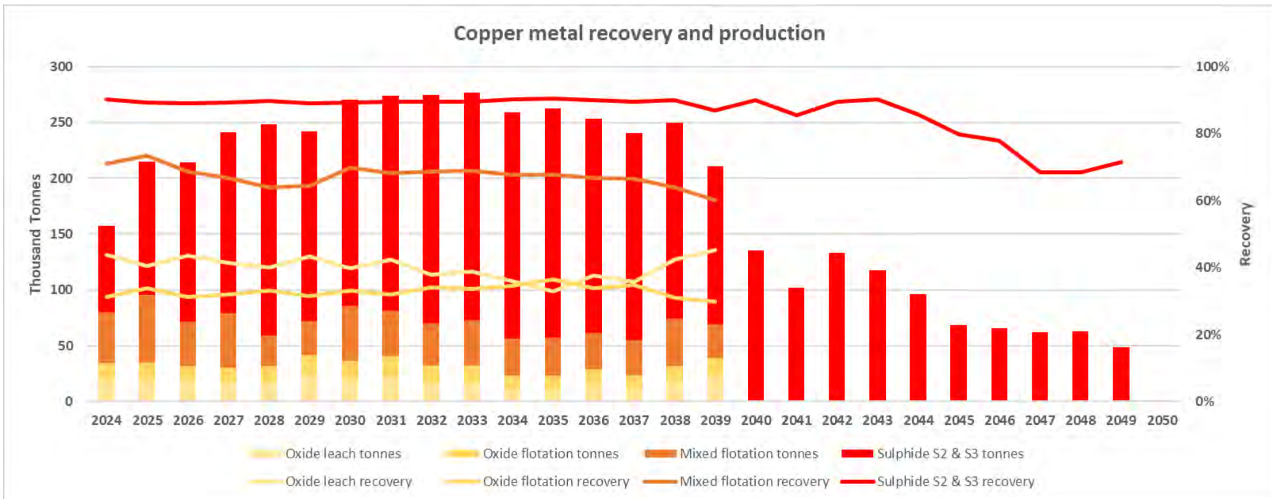


Figure 16-21 Kansanshi LOM schedule, annual oxide stockpile tonnes

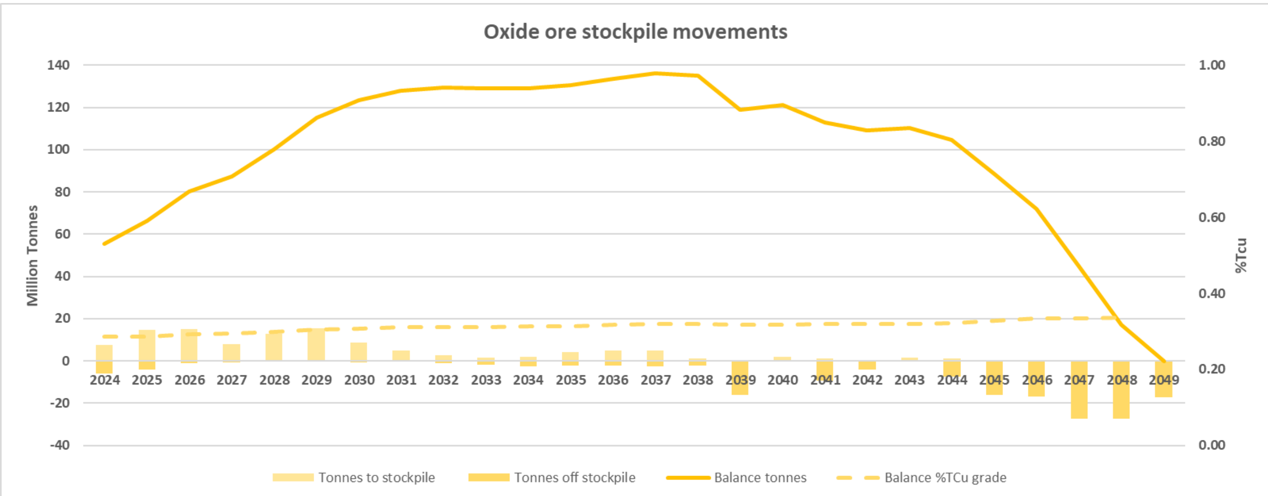


Figure 16-22 Kansanshi LOM schedule, annual mixed stockpile tonnes

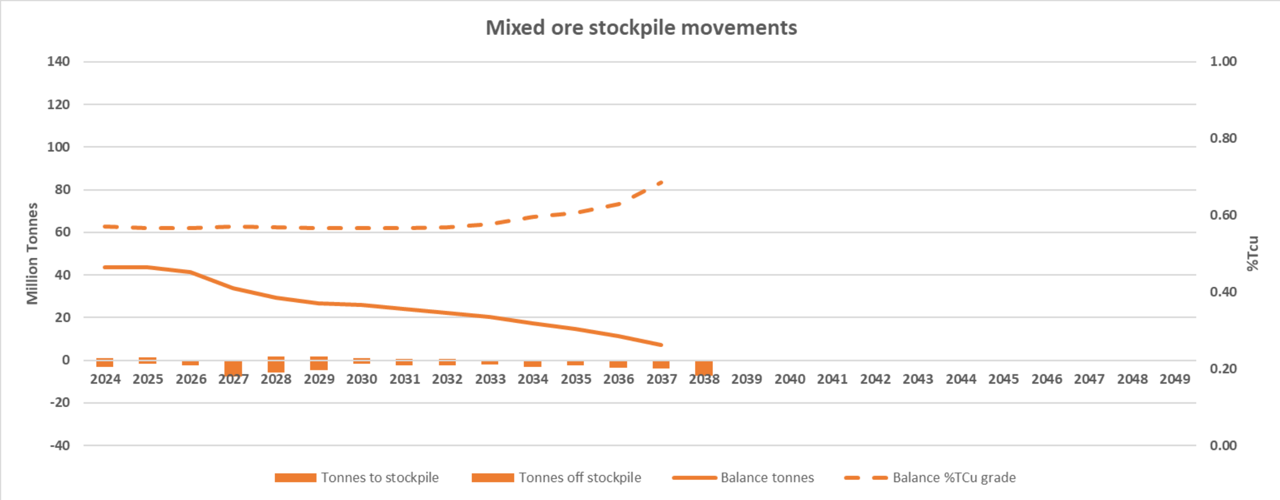
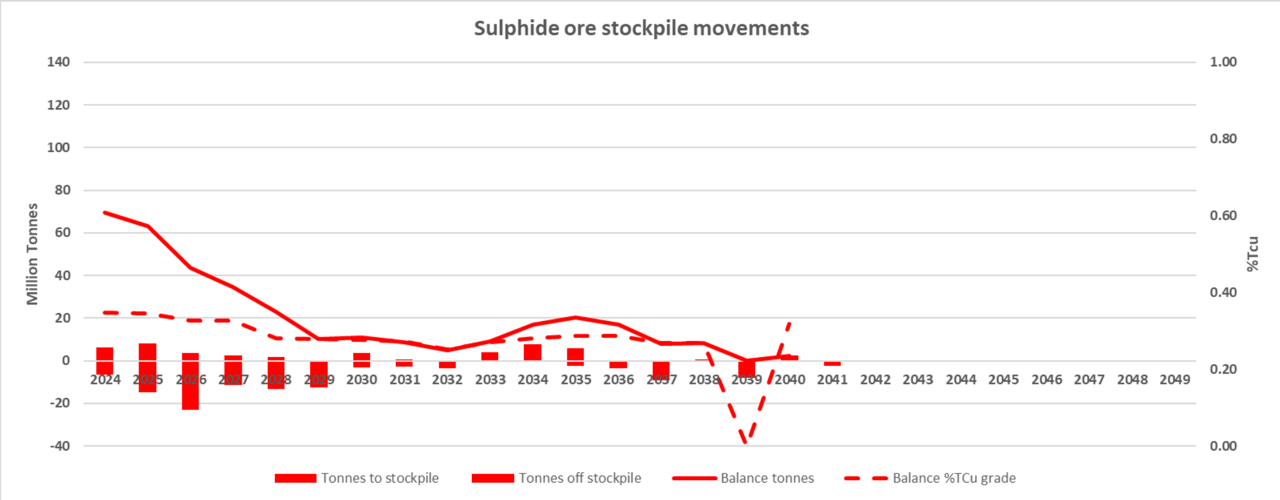


Figure 16-23 Kansanshi LOM schedule, annual sulphide stockpile tonnes



In order to consider infrastructure and longer term equipment requirements, a production schedule was also completed inclusive of Inferred Resource, mined and processed after depletion of the Mineral Reserve inventory. This indicative schedule shows that the mine life could be extended to 2049, with processing to 2053.

16.8.2 Accounting for additional gold recovered

Historically, the Kansanshi processing circuits have always recovered more gold than would be expected from the modelled insitu gold grades and from the estimated gold in the Mineral Reserve inventory. This "additional gold" is typically recovered from gravity concentrators in each of the plant circuits, and also, as gold recovered into anode.

As an adjunct to the recovered gold associated with the formal Mineral Reserve estimate, this additional gold can be approximated by projections based on historical performance. The additional gold estimated in this way is listed in Table 16-14.

Relative to the historical gold production described in Item 14.13, the proportion of future payable gold recovered additional to that associated with the Mineral Reserve inventory, is in the order of 45%.

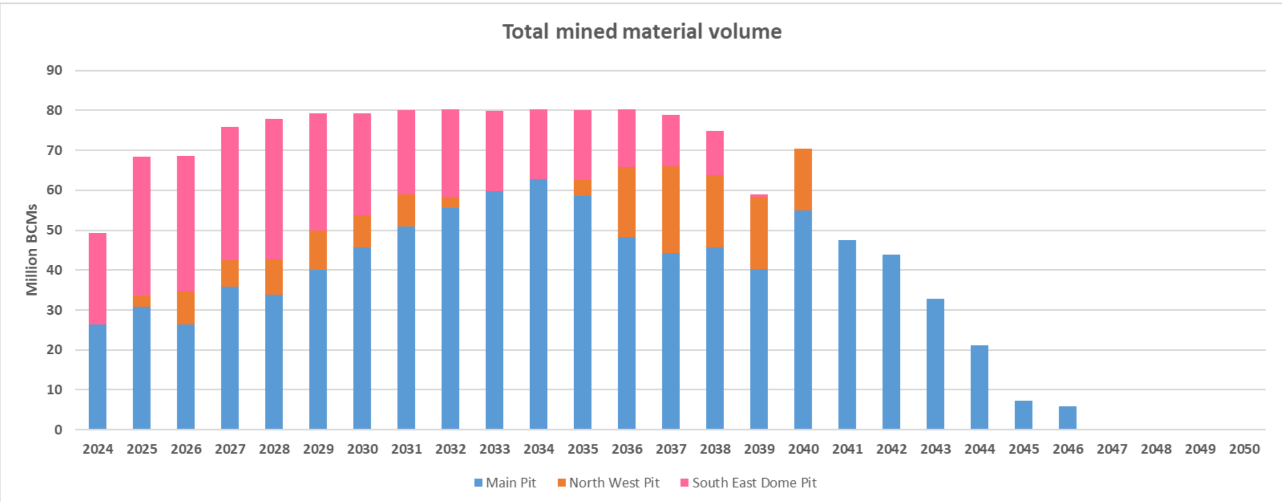
Table 16-14 Kansanshi LOM schedule, recovered gold additional to the Mineral Reserve

Year	Recovered metal Mineral Reserve (kt Cu)	Recovered metal Mineral Reserve (koz Au)	Payable gold Reserve (koz Au)	Payable gold additional (koz Au)	Payable gold total (koz Au)
2024	157.6	49.2	44.1	27.6	71.7
2025	215.1	70.7	63.7	28.4	92.1
2026	213.7	62.4	56.2	55.6	111.7
2027	241.9	64.5	58.0	61.8	119.9
2028	248.9	62.4	56.2	64.0	120.2
2029	242.2	62.2	55.2	43.3	98.6
2030	270.8	66.5	59.3	53.6	112.9
2031	273.8	62.9	55.6	50.2	105.9
2032	274.6	72.1	64.1	53.8	117.9
2033	276.8	73.7	65.8	55.7	121.5
2034	259.3	63.4	56.0	45.7	101.8
2035	261.4	64.8	57.4	47.3	104.7
2036	253.6	61.0	53.9	41.8	95.7
2037	240.5	65.1	57.5	42.8	100.2
2038	249.6	69.2	61.3	53.3	114.7
2039	211.3	59.1	51.6	35.1	86.7
2040	135.6	38.3	33.9	31.4	65.3
2041	102.3	33.9	29.9	20.7	50.5
2042	133.5	37.5	33.2	30.4	63.6
2043	117.2	34.5	30.3	24.3	54.6
2044	96.0	29.7	25.8	16.8	42.5
2045	68.4	29.2	25.6	11.4	37.0
2046	65.8	32.9	29.2	12.2	41.5
2047	62.5	21.9	18.5	8.0	26.5
2048	62.6	22.0	18.6	7.9	26.5
2049	41.2	14.5	12.3	6.4	18.6
2050					
	4,776.3	**1,323.7**	**1,173.3**	**929.6**	**2,102.9**

16.8.3 Mining sequence

Figure 16-24 shows the mining sequence and production (ore plus waste mining volume) profile. Mining in Main Pit continues until 2046, whereas mining in the North West Pit ceases in 2040. Mining in the South East Dome Pit proceeds until 2039.

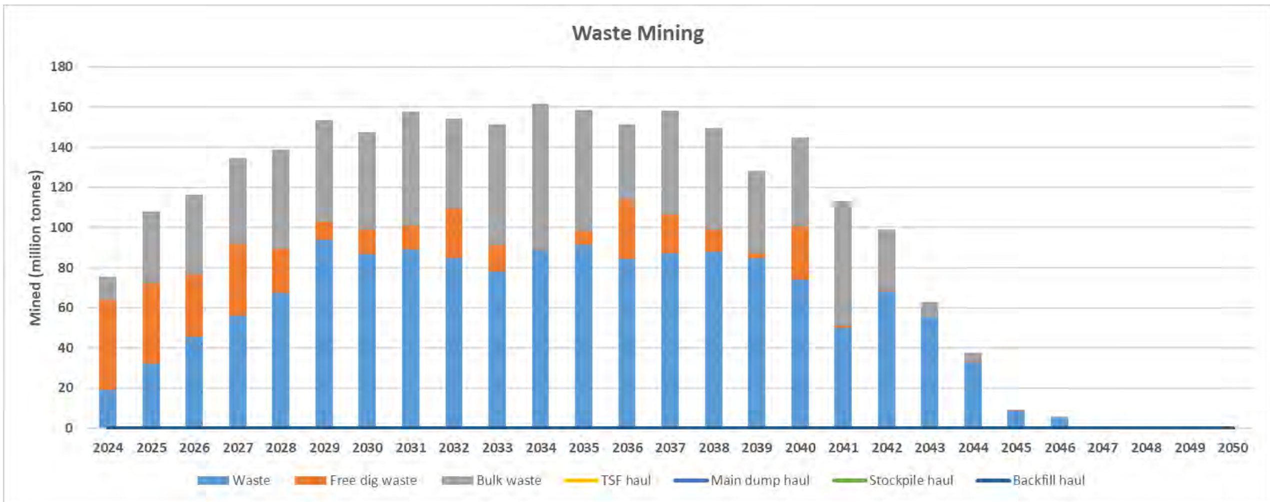
Figure 16-24 Kansanshi LOM schedule, mining sequence



16.8.4 Waste dumping schedule

Figure 16-25 shows the life of mine waste dumping schedule.

Figure 16-25 Kansanshi LOM schedule, waste dumping volumes sequence



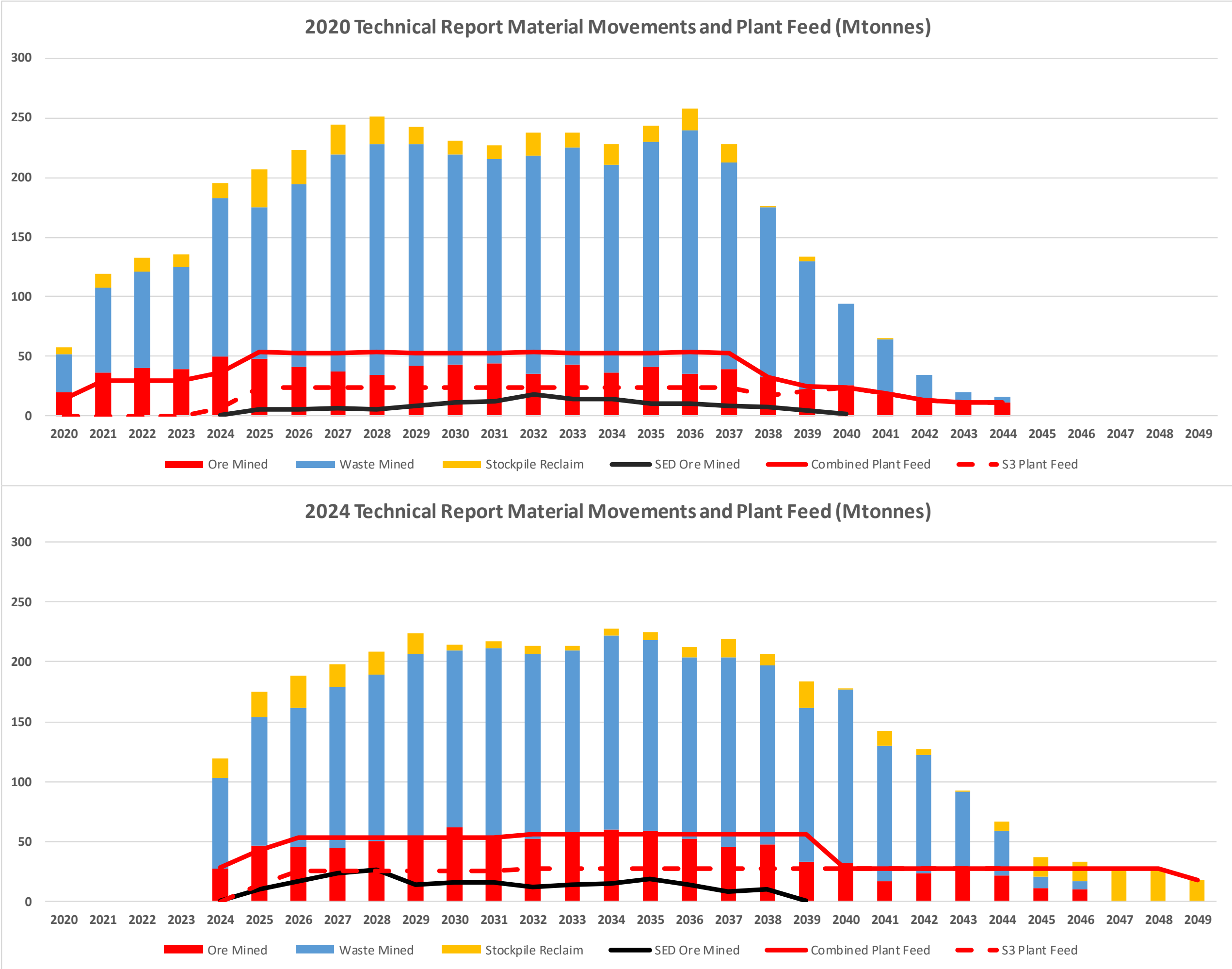
16.8.5 Comparison with 2020 Technical Report schedule

Figure 16-26 provides a comparison chart between the scheduled total material movement (and plant feed) profile provided in the 2020 Technical Report and the equivalent profile from the above described schedule. Some observations of note are:

- in 2020, 183 Mt of total material movement (excluding stockpile reclaim) was scheduled for 2024, rising to a first peak of ~230 Mt in 2028/29 and a second peak in 2036
- whereas in this current schedule, 104 Mt of total material movement (excluding stockpile reclaim) is scheduled for 2024, rising to a lesser peak of 212 Mt in 2031 and then to 222 Mt in 2034
- in 2020, mining at South East Dome (a then smaller Reserve tonnage pit) was scheduled to commence in 2024, rising gradually to an ore and waste peak of 18 Mt in 2032, and then ceasing in 2040
- whereas in this current schedule, mining at South East Dome (a now larger Reserve tonnage pit) also commences in 2024, but rises faster to an ore and waste peak of 26 Mt in 2028, and then ceases in 2039
- in 2020, the S3 plant was scheduled to commence processing in the second half of 2024, with a small feed contribution from South East Dome and from stockpile reclaim
- whereas in this current schedule, the S3 plant is scheduled to commence processing in the second quarter of 2025, with a more substantial contribution from South East Dome and from stockpile reclaim

Further information on actual vs scheduled performance for the period 2021 to 2023 is provided in Item 6.6.

Figure 16-26 Kansanshi LOM schedule, comparison against the 2020 Technical Report schedule



16.9 Primary mining equipment

The Kansanshi mining fleet comprises diesel powered production drills, various sized hydraulic shovels/excavators and a variety of haul truck makes and capacities. Mining is carried out using the Company's own equipment fleet. The existing primary mining fleet inventory is listed in Table 16-15.

To accommodate the increased volume associated with the S3 Project, an expanded mining fleet is required. The purchase of this fleet has already commenced. The expanded mining fleet will feature similar ultra-class equipment as at the Company's other large scale operations. The operations will also benefit from new electrical loading and drilling equipment along with the extension of the current electric trolley assist infrastructure. The additional fleet that has been determined as required to cater for expanded production, and for replacement, is also listed in Table 16-15. Included in the additional fleet required from 2024, are items specifically procured to cater for S3, as follows:

- Epiroc STR45 drills; 8 of 15 required are for S3
- Epiroc Pitviper 231E drills; 6 of 6 required are for S3
- Hitachi EX5600-7E shovels; 6 of 6 required are for S3
- Hitachi EH4000 trucks; 36 of 43 required are for S3
- Komatsu D375A dozers; 6 of 12 required are for S3
- Komatsu D475A dozers; 2 of 5 required are for S3

Table 16-15 Kansanshi Operations, existing and additional primary mining equipment fleet

Fleet Item	Make/Model	At December 2023	From January 2024
Drills	Drill Tech D25KS	16	14
	Pantera DP1500i	10	8
	Epiroc SmartROC T45-10LF	2	-
	Epiroc STR45-10LF	3	15
	Epiroc DM30	-	2
	Epiroc Pitviper 231E	-	6
Shovels/excavators	Hitachi EX2500 BE	-	-
	Hitachi EX5600-7E (550 t)	-	6
	Liebherr R9250D BH (250 t)	3	3
	Liebherr R9350D BH (350 t)	8	8
	Liebherr R9350 ER (330 t)	4	3
	Liebherr R9400E (400 t)	-	1
	Liebherr R9100D (100 t)	-	2
	Liebherr R984C (120 t)	6	4
Trucks	Caterpillar 777D	-	-
	Caterpillar 785D (150 t)	56	47
	Hitachi EH3500ACii (170 t)	38	36
	Hitachi EH3500AC3 (170 t)	2	2
	Hitachi EH4000AC3 (220 t)	10	43
Wheel loaders	Caterpillar 992G	-	-
	Hitachi LX290E	-	-
	Komatsu WA500-6	2	-
	Komatsu WA900-3	1	-
	Komatsu WA900-8R	1	2
	Komatsu WA1200-6	1	-
Track/wheel dozers	Komatsu D275A-5R	2	2
	Komatsu D375A-5/6	9	12
	Komatsu D475A-5	3	5
	Komatsu D475A-8R	1	-
	Liebherr PR764	7	7
	Komatsu WD900-3	4	4

In addition to the current equipment listed in Table 16-15 ancillary mining equipment includes:

- smaller excavators
- integrated tool carriers
- graders
- water carts
- cable reeler
- tyre handlers
- stemming loader
- fuel and service trucks

ITEM 17 RECOVERY METHODS

17.1 Mineral processing methods

The current processing facilities at Kansanshi comprise three main processing circuits; an oxide circuit with an approximate capacity of 7 Mtpa, a mixed ore circuit with a capacity of 8 Mtpa and the S2 sulphide circuit with a capacity of 13 Mtpa.

All ore types are treated in separate circuits via crushing, milling and flotation to produce copper in concentrate. In addition, oxide ore and a portion of mixed ore flotation tailings are leached and subject to solid-liquid separation, followed by solvent extraction (SX) and electrowinning (EW) to produce copper cathode.

Key production data for the last five years are listed in Table 17-1.

Table 17-1 Kansanshi production figures, 2019 to 2023

Year	Units	2019	2020	2021	2022	2023
Oxide Feed						
Tonnes processed	Mt	7.20	7.44	7.16	7.87	7.23
Head grade	%Cu	1.12	0.93	0.72	0.57	0.82
Recovery	%	81.5	75.8	68.8	63.5	75.6
Mixed Feed						
Tonnes processed	Mt	7.70	8.17	7.60	7.71	7.77
Head grade	%Cu	1.05	1.00	0.96	0.63	0.63
Recovery	%	77.4	81.1	82.3	74.3	70.7
Sulphide Feed						
Tonnes processed	Mt	12.91	13.53	13.39	13.16	12.45
Head grade	%Cu	0.89	0.83	0.88	0.71	0.51
Recovery	%	91.2	92.0	91.3	89.5	87.6
Total Feed						
Tonnes processed	Mt	27.81	29.13	28.14	28.74	27.45
Head grade	%Cu	0.99	0.90	0.86	0.65	0.63
Recovery	%	84.2	84.1	83.3	78.8	78.6
Gold Recovery						
Head grade	g/t Au	0.20	0.20	0.20	0.20	0.14
Gold in concentrate	kg	2,902	2,785	2,883	2,477	1,465
Gold in dore	kg	1,620	1,209	1,105	932	681
Total gold produced	oz (troy)	145,385	128,409	128,198	109,617	68,975
Overall gold recovery	%	80.2	70.0	71.1	59.3	55.0

Figure 17-1 is a recent aerial photograph of the processing facilities at Kansanshi. The current Kansanshi flowsheet for the concentrator circuit is shown in Figure 17-2, whilst Figure 17-3 shows the hydrometallurgical process flowsheet.

Figure 17-1 Aerial photograph of Kansanshi processing facilities (source: FQM)

Figure 17-2 Kansanshi concentrator, process flow diagram (source: FQM)

(yellow is sulphide feed, red is mixed feed and green is oxide fed)

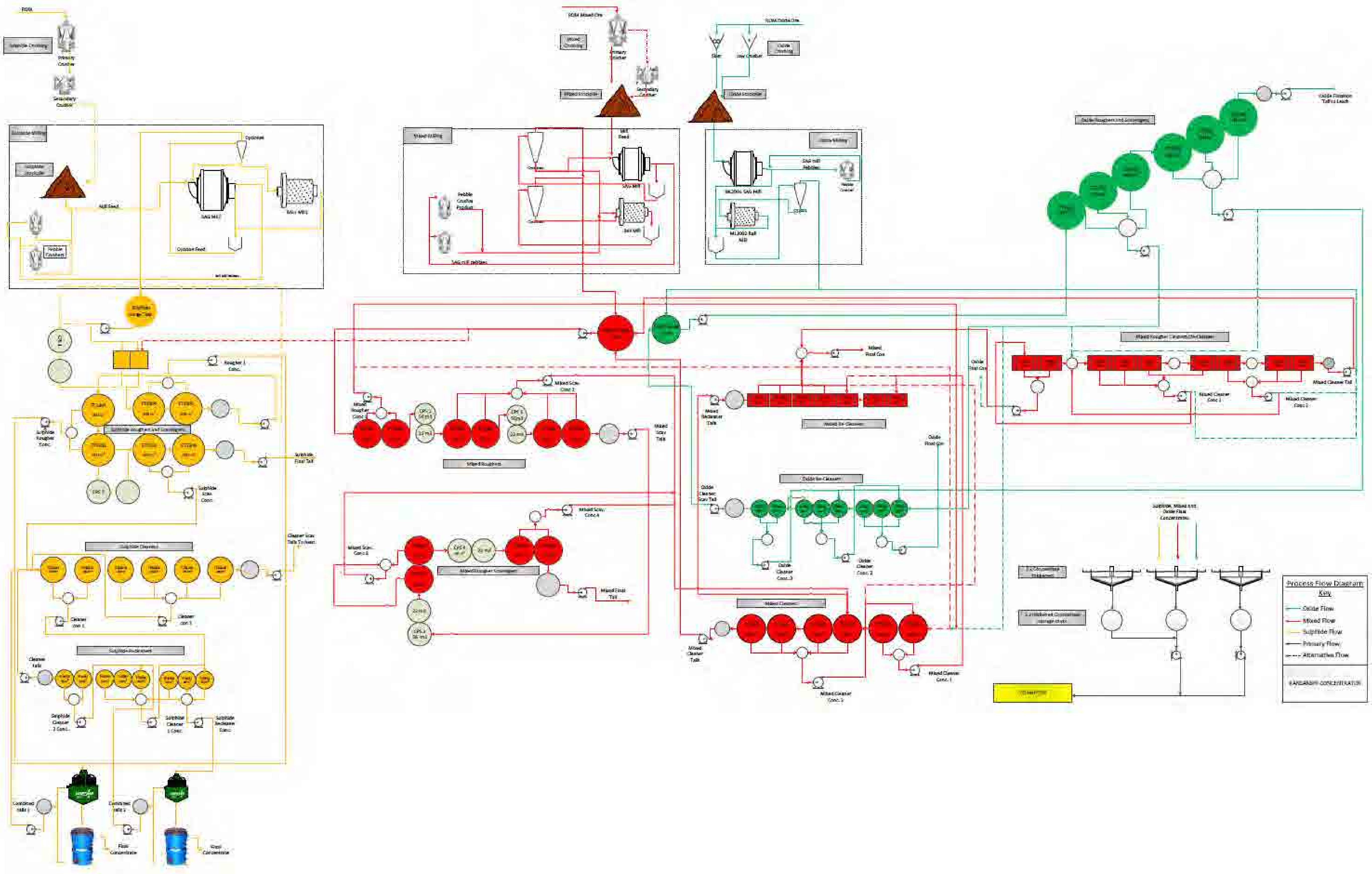
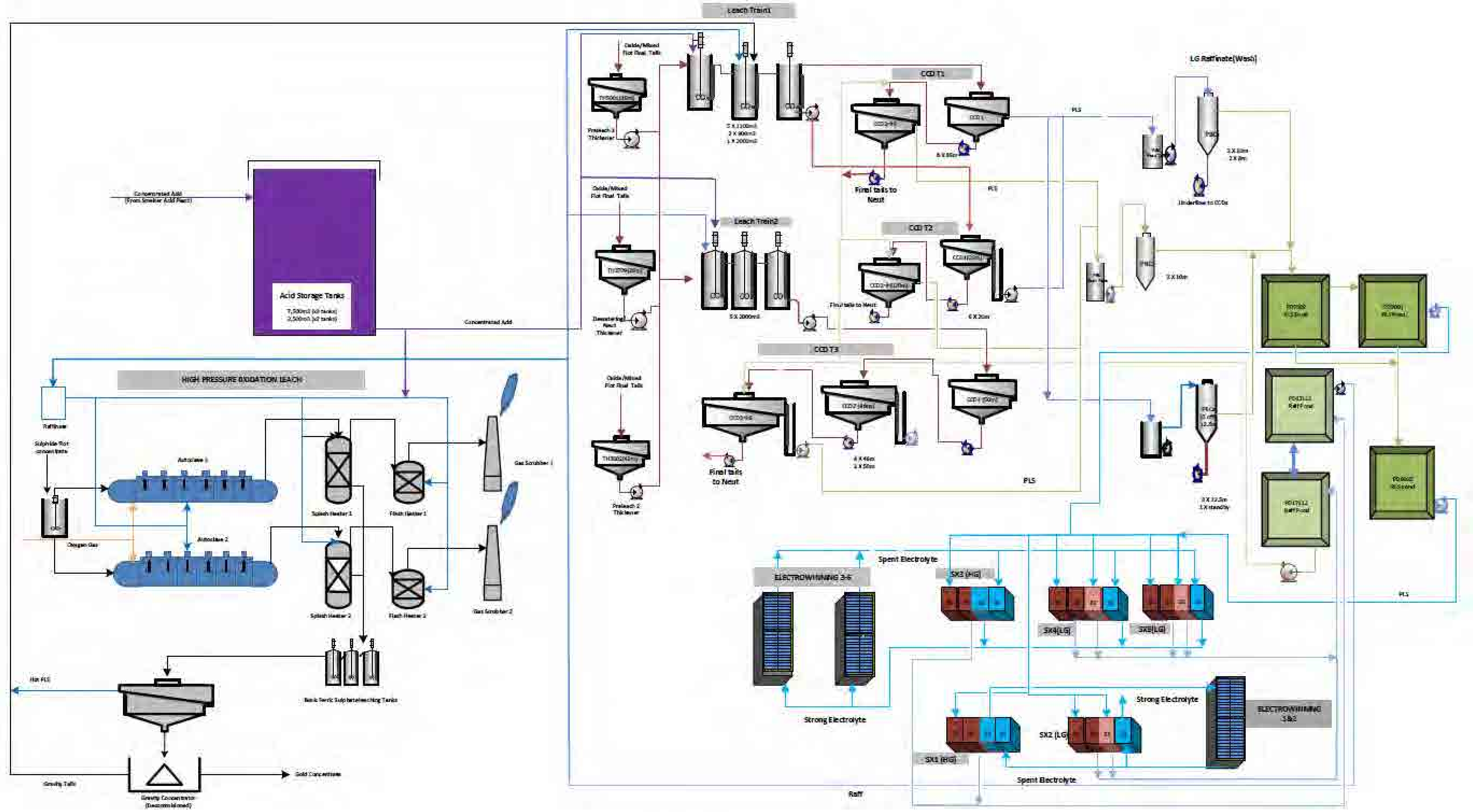


Figure 17-3 Kansanshi hydrometallurgical plant, process flow diagram (source: FQM)



Date: 26/02/2024

17.1.1 Ore delivery, primary crushing and crushed ore storage

Direct mined/tipped ore is hauled to a surface run-of-mine (ROM) pad located immediately to the north of the processing plant. Feed from direct mine sources, in addition to ore reclaimed from a number of surface stockpiles is tipped into one of three separate primary crusher dump pockets.

The sulphide and mixed ore crushing circuits comprise primary gyratory and open circuit secondary cone crushers, whilst the oxide crushing circuit comprises a primary jaw crusher, a secondary sizer and a semi-mobile jaw crusher. Each of the crushed ore product streams is conveyed to dedicated coarse ore stockpiles ahead of the grinding circuits.

17.1.2 Grinding circuits

The grinding circuits are all SABC circuits, each comprising a SAG mill, ball mill and pebble **c**rusher.

Each mill is equipped with hydrocyclones, with cyclone overflow gravitating to the respective rougher flotation circuit, and cyclone underflow from both the SAG and ball mill cyclones being directed to the ball mill. Coarse material from the SAG mill discharge screen is conveyed to a pebble crusher for each circuit and crushed pebbles are returned to the SAG mill feed conveyor

Each circuit is capable of gravity gold recovery from multiple centrifugal concentrators, mainly treating a bleed stream from the cyclone underflow in each circuit. The various gravity concentrates are all treated in the central gold room facility to produce gold doré bars.

17.1.3 Flotation

Each of the circuits includes a flotation section to recover sulphide minerals from the different ore types. Each flotation circuit includes rougher, rougher-scavenger, cleaner and recleaner sections, and the concentrate from the sulphide circuit is treated in primary and secondary Jameson flotation cells and upgraded in columns to produce a final concentrate product. The oxide and mixed ore circuits use controlled potential sulphidisation (CPS) and NaHS to improve the recovery of secondary and partially oxidised minerals. CPS is a recent enhancement to the sulphide flotation circuit, implemented to improve recoveries.

All of the flotation concentrates are thickened and then pumped to the smelter for filtration prior to smelting.

A new cleaner flotation circuit was commissioned in early 2022 to treat the sulphide concentrates. This circuit comprises six 150 m^3 cells as first cleaners and cleaner scavengers, with the first cleaner concentrates being re-cleaned in the existing cleaner circuit. Final concentrates are upgraded in two new flotation columns.

The new bank of first cleaner cells provides increased residence time and allows the circuit to be opened up and a cleaner scavenger tail to be discarded to final tails. The extra cleaning stages and the new columns produce a higher grade concentrate than was being produced, with lower levels of carbon and pyrite. Higher grade concentrates lead to increased concentrate treatment rates at the smelter.

This circuit is now being modified to handle S3 concentrates. Rougher concentrates will be pumped from the S3 site to this circuit in the main plant. A Jameson cell will handle the higher grade rougher concentrates, and a series of six x 300 m^3 cells will act as first cleaners for the S3 rougher scavenger concentrates and the S2 rougher concentrates. The existing 6 x 150 m^3 cells will be reconfigured as second and third cleaners, and the original cleaner circuit comprising eight x 30 m^3 cells, installed in 2008, will be de-commissioned. Additional columns will be installed to handle the increased concentrate production.

Concentrates produced from these circuits will be thickened in the existing concentrate thickeners at Kansanshi and pumped to the smelter through the existing pumps and pipelines.

17.1.4 Acid leaching, CCD, SX and EW

The flotation tailings stream from the oxide ore circuit is acid leached in a circuit that comprises five leach tanks followed by a counter-current decantation (CCD) train of six thickeners. The liquor from the CCD overflows is treated through pinned bed clarifiers to remove suspended solids and then collected in a common pregnant leach solution (PLS) pond.

High pressure leach (HPL) autoclaves are used to treat a portion (up to 100,000 tpa) of the flotation concentrate. The product from the autoclaves is added to the leach circuit to provide additional heat and ferric ion to assist in leaching.

PLS is treated through SX and EW to produce copper cathodes for export. Raffinate from the SX circuit is returned to the leach for re-use of the acid. There are five SX trains and three EW tank houses, although only two SX trains and one EW tank house are currently being used.

The flotation tails from the mixed ore may be acid leached, or discharged to final tailings, depending on the tailings grade and Gangue Acid Consumption (GAC) of the ore, the availability of acid from the smelter, and the current acid price. These factors may result in the excess acid being sold to external customers rather than being used to leach the mixed ore flotation tailings. Mixed ore flotation tailings are treated in a leach, CCD, clarifier circuit similar to that for the oxide flotation tailings. The final PLS from both circuits is combined prior to final copper recovery.

The mixed ore circuit can also be used to treat sulphide ore in the event that there is an imbalance of sulphide and mixed ore production from the mine.

17.1.5 Gravity gold recovery

Gold production occurs through natural flotation of finer gold particles with the chalcopyrite. The gold present in the flotation concentrate ultimately reports to the copper anodes exported from the smelter.

Gold recovery is enhanced with the installation of centrifugal concentrators in the milling circuits. Concentrators were initially installed in the sulphide milling circuits, S1, which became the mixed ore circuit, and S2, followed by the oxide milling circuit in 2018.

Gravity concentrates are treated in a central facility that was installed in 2010. The concentrate is upgraded on rougher and cleaner shaking tables, followed by acid treatment for removal of residual sulphides. Acid treated concentrates are washed, dried and smelted in an induction furnace to produce doré bars.

The gravity gold circuits have been subject to continuous improvement through the installation of additional and more efficient concentrators throughout the circuits.

17.1.6 Neutralisation of acid leach tailings

The acid leach tailings are neutralised with sulphide flotation tailings to achieve a minimum pH of 5.5 prior to final tailings disposal. Lime addition may be used in neutralisation if necessary (when the sulphide flotation circuit is off line for example). This also influences the decision to leach the mixed ore flotation tails, as the overall neutralisation balance requires approximately 60% sulphide float tails in the final tailings.

17.1.7 Tailings storage facilities

There are currently two licenced operating tailings storage facilities (TSFs). TSF1 is a cross-valley type dam sited at the head of a small tributary stream inside of the mining licence area, and covering an area of approximately 6.5 km^2. This dam was originally designed to provide sufficient tailings storage capacity for the first sixteen years of mine life, and has been periodically upstream raised with cyclone tailings. Up to December 2023, tailings deposition into TSF1 had amounted to 245 million tonnes.

TSF2, a second cross valley dam was commissioned in 2012. It has also been upstream raised with cyclone tailings. By the end of 2023, tailings deposition into TSF2 had amounted to 172 million tonnes.

These facilities have adequate combined capacity for future tailings generated from the expanded processing circuits. However, potential sites for a third tailings storage facility are being evaluated. A third TSF will reduce the average rates of rise on the tailings dams and improve operating flexibility.

17.1.8 Treatment of flotation concentrates

Until 2014, concentrate produced from the three circuits was filtered (with pressure filters) and sold to off-site smelters within Zambia.

A leach capacity expansion at Kansanshi in 2014 coincided with the construction of an on-site smelter with a capacity to treat 1.2 million tpa of concentrate and produce 300 ktpa of blister copper (anode).

The rationale behind the new smelter at Kansanshi arose from the then current shortage of smelting capacity in Zambia (exacerbated as the Company's Sentinel Project came on line) potentially constraining the sale of concentrate from the proposed S3 plant.

The smelter produces large volumes of sulphuric acid as a by-product, which can 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 Kansanshi smelter now receives essentially all of the thickened and filtered flotation concentrate (in addition to some of the concentrate from the Sentinel plant), from which anode copper is produced and sold.

17.1.9 Water supply

The main sources of processing water are from the Main Pit dewatering shaft, the new Kansanshi Dam and the Solwezi River, and return water from the tailings dam, the sedimentation ponds and the smelter. Potable water comes from a dedicated borefield.

Total make-up water requirements are 3,450 m^3/h, or 82,800 m^3/day.

17.2 The S3 Expansion Project

The Kansanshi S3 Project is an expansion to the existing sulphide processing facilities. The expansion includes the construction of a stand-alone copper concentrator capable of treating an additional 27.5 Mtpa of sulphide ore, and an overland conveyor from a near-mine surface transfer bin receiving crushed ore from the primary crushers.

The S3 Project has been planned for some time but had been deferred in recent years. Most of the engineering design (with the exception of the electrical design) had been completed, but the flotation circuit design has now been updated based on more recent operating experience at Sentinel, Kansanshi, and Cobre Panama to include CPS and additional concentrate cleaning capacity to handle higher levels of carbon and pyrite in the ore feed.

The Project is currently under construction, with first ore expected to be fed to the mills in Q2 2025. Figure 17-4 is a recent photograph showing the construction progress on the site.

Figure 17-4 Kansanshi S3 plant, partially constructed

The 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, and which have been operating consistently and efficiently since they were commissioned.

Details on specific flowsheet development and specific process design aspects are summarised below, with a Process Flow Diagram being presented in Figure 17-5.

Figure 17-5 shows a general arrangement plan for the S3 plant.

17.2.1 Flowsheet

The mechanical design for the S3 plant is based on the design principles developed for the Company's Sentinel concentrator, with S3 utilising one milling circuit rather than the two at Sentinel. This allows commonality of design and equipment. Improvements to the Sentinel design, and experience gained at the Company's 100Mtpa Cobre Panama operation have been incorporated into the S3 design.

The S3 metallurgical design is based upon the current Kansanshi sulphide circuit as this has been proven on the ore types to be treated. Sulphide ore samples from the South East Dome deposit (which is yet unmined) appear to be geologically and mineralogically similar to the carbonaceous phyllite ore mined in the existing Main Pit.

17.2.2 Plant throughput

A revised plant throughput rate of 27.5 Mtpa has been nominated for the S3 design. Based on 8,000 operating hours per year, this is equivalent to a mill feed rate of 3,438 tph. At a design nominal head grade of 0.58% Cu and a design nominal 90.7% recovery, this feed rate equates to approximately 145,000 tpa copper production at 24 to 26% concentrate grade.

Figure 17-5 Kansanshi S3 plant, Process Flow Diagram (source: CPC)

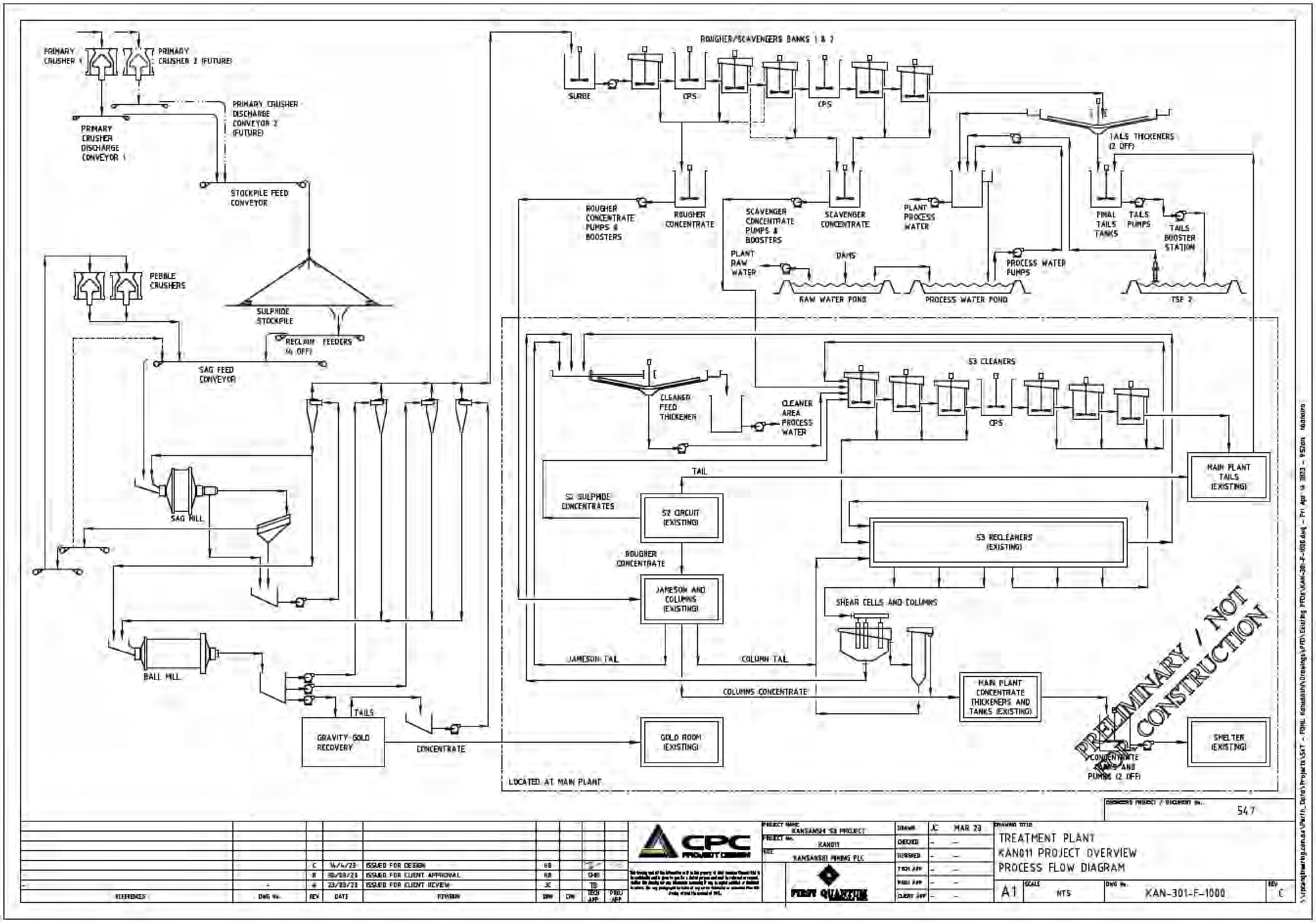
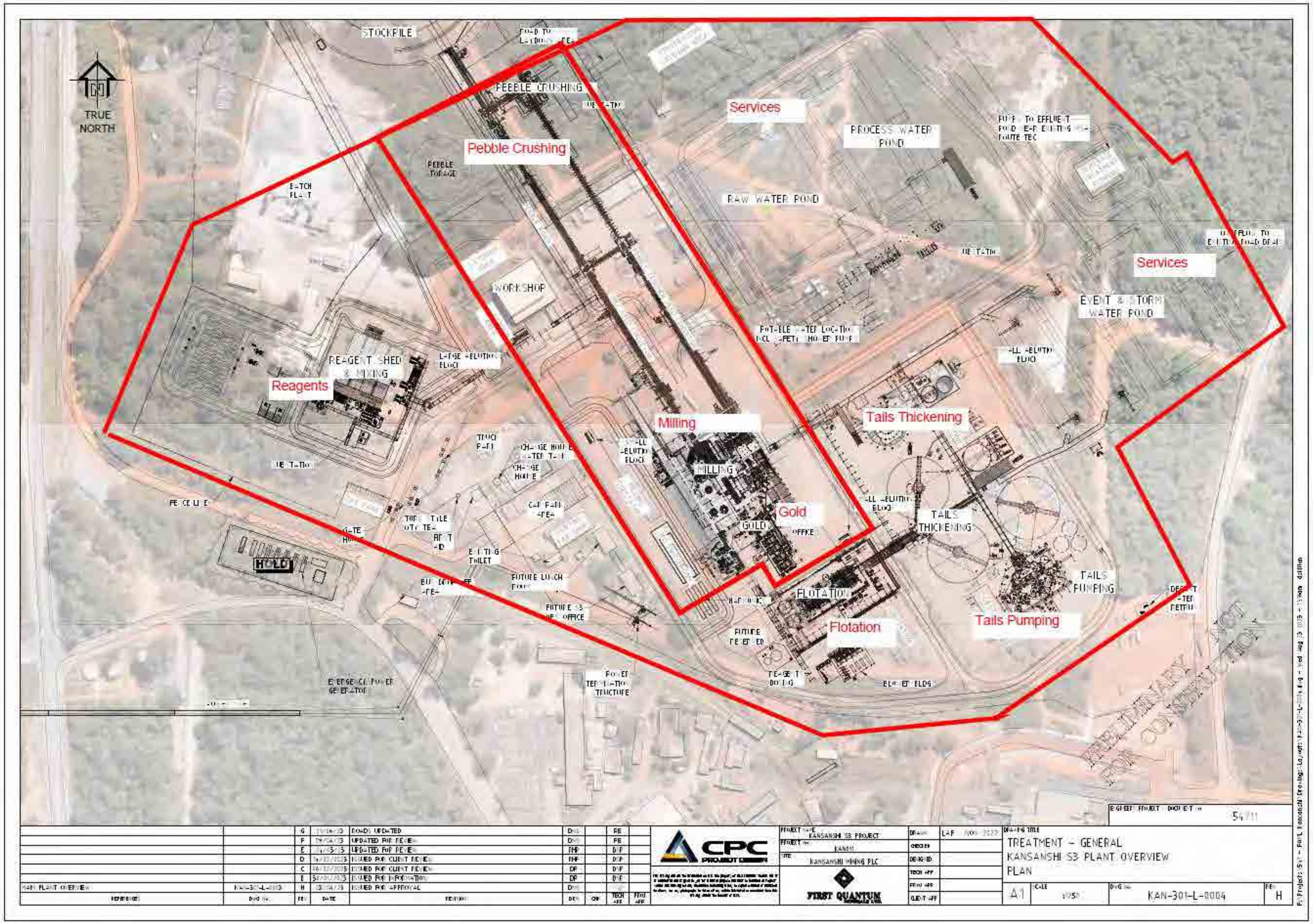


Figure 17-6 Kansanshi S3 plant, general arrangement plan (source: CPC)



17.2.3 Process design criteria

The key process design criteria for the S3 expansion are listed in Table 17-2.

Table 17-2 S3 sulphide plant, process design criteria (from CPC Engineering, 2023)

Parameter	Value
Annual Throughput	25 Mtpa (now revised to 27.5 Mtpa)
Copper Head Grade (Nominal)	0.58%
Gold Head Grade	0.15 g/t
Concentrate Copper Grade (Nominal)	24 to 26%
Copper Recovery (Nominal)	90.7%
Crushing Plant Availability	6,570 h/annum (75%)
Concentrator Availability	8,000 h/annum (91.3%)
Crusher Operation	18 h/day
Primary Crusher (one initially operating)	3,805 dtph
Mill Circuit Feed Rate	3,125 dtph
Final Concentrate Make (at 24% Cu)	548,000 dtpa (603,000 dtpa)

17.2.4 Primary crushing and crushed ore storage

Primary crushing will be carried out by two gyratory crushers (PCs), one installed initially, and the second in future when the pit is developed. Both crushers will be installed near-surface, with the first being on the south western side of the Main Pit, and the second on the southern edge of the Main Pit (refer to Item 16.2.3).

Each crusher will be equipped with a transfer conveyor and a variable length discharge conveyor connecting the PC facility to a surface transfer point (Figure 17-5). The mine plan attempts to minimise the number of in-pit transfer points along the discharge conveyor route. At the pit top transfer point, the crushed product will be transferred onto an overland conveyor leading to the S3 plant stockpile.

Crusher operation will be for nominally 18 hours per day (availability of 75%) requiring an average feed rate to the crushed ore stockpile of 3,805 tph. The crushed ore stockpile will have 13 hours live capacity, equivalent to about 40,000 tonnes. Ore will be recovered from below the crushed ore stockpile by four variable speed apron feeders located in a single reclaim tunnel.

17.2.5 Grinding circuit

Crushed ore reclaimed from the stockpile will be fed to a 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 the ground product then pumped to a dedicated cyclone pack.

Ball milling will be accomplished in a single 22 MW, 8.5 m diameter, 13.3 m EGL (effective grinding length) ball mill. It 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 enable mill liner changing.

17.2.6 Gold circuit

Gravity concentrators will be installed in the milling circuit to recover coarse gold. The gravity concentrates will be trucked to the gold room in the main plant, where they will be repulped and either treated in the existing gold treatment facility (which will require additional equipment), or they will be pumped to the final flotation concentrate thickener and then to the smelter.

17.2.7 Flotation

Two banks of flotation cells will be used for rougher/scavenger duty. Each bank will be headed by a single 600 m^3 tank cell followed by 4 x 300 m^3 cells arranged as 2 stages of CPS (controlled potential sulphidisation).

CPS enhancement has been included in the sulphide flotation circuit to improve recoveries, particularly of partially oxidised sulphides and secondary sulphide minerals. This enhancement has been incorporated into the existing sulphide flotation circuit at Kansanshi (S2).

Two concentrate streams (first rougher concentrates and rougher scavenger concentrates) will be pumped to a cleaner circuit at the main processing plant. This cleaner circuit will be integrated with the new cleaner circuit recently commissioned for S2 concentrate.

Two Jameson cells will upgrade the first rougher concentrates, which will then be cleaned in columns before being sent to final concentrates. The scavenger concentrates will be upgraded in a bank of mechanical cells, followed by two Concorde cells and two columns to produce final concentrates for smelter feed.

17.2.8 Concentrate handling

Final concentrates will be dewatered in the existing concentrate thickeners which are adequate for the future concentrate production. However all six underflow pumps will be upgraded to handle the increased duty.

The concentrate thickener underflow will be pumped to filter feed tanks acting as surge capacity between the flotation and filtration circuits. Two new concentrate tanks will be added to supplement the two existing agitated stock tanks. Three sets of concentrate transfer pumps (one new set plus the two existing sets) will pump the concentrate to the smelter.

Concentrate slurry will be filtered using four Larox filters (one new filter added to supplement the existing three units) located adjacent to the smelter. Filtered concentrate will report either to the smelter feed conveyor or to a conveyor feeding to a storage shed.

17.2.9 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.

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.

Main plant tailings (at 30 Mtpa) will be added to S3 tailings (27.5 Mtpa) in the tails surge tank, and the combined tailings at 55Mtpa will be pumped to TSF 2.

Two duty tails lines, each with three stages of centrifugal pumps plus a booster stage will be provided for tailings pumping. A third line of pumps will be provided as a common standby.

The tailings storage facility (TSF) is an existing dam (named TSF2) currently used by the existing Kansanshi operation. For the first years 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.

A decant system will be added to TSF 2 to enable water to be recovered from the tailings after placement. The water will be returned to the process water tank in the S3 process plant.

A third tailings storage facility is currently under consideration, with several sites being evaluated. A third TSF will improve operational flexibility and reduce the rates of rise on the current dams.

17.2.10 Process water demand

Process water demand will expand to 118 ML/day once the S3 plant is completed.

A new fresh water dam with a capacity of 3.3 mm^3 was previously constructed for the S3 Project and is already full and supplying water to the current operations.

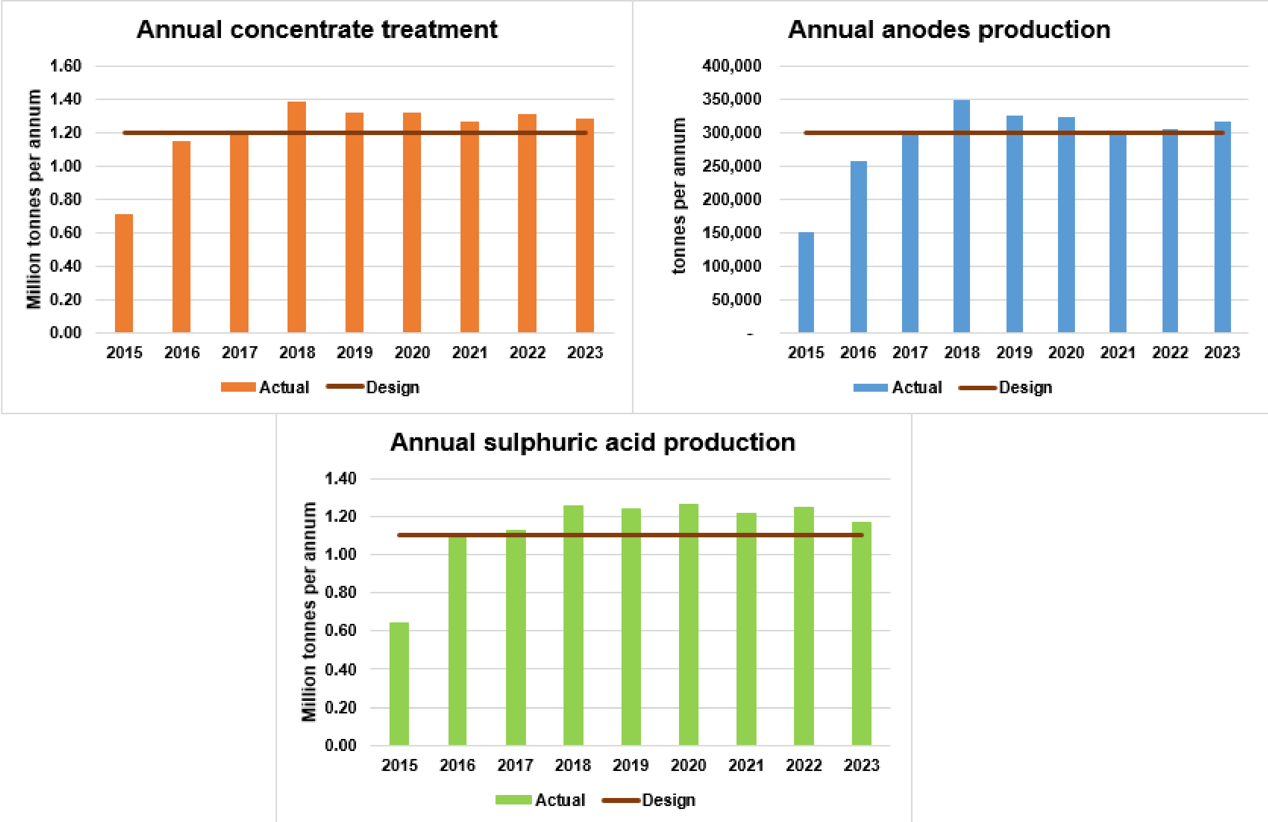
17.3 Kansanshi smelter

The Kansanshi copper smelter (KCS) began operation in March 2015. The smelter features a single Isasmelt furnace for smelting, one 6-in-line electric furnace for matte settling and slag cleaning duties, four Peirce-Smith converters (PSCs) and two conventional rotary anode furnaces. Figure 17-6 is an aerial view of the smelter installation. Figure 17-7 provides the key smelter production data since start-up.

A 1,450 t/day oxygen plant and a 4,000 t/day sulphuric acid plant are an integral part of the smelting operations.

Figure 17-6 Aerial view of the Kansanshi smelter

Figure 17-7 Smelter production data since start-up



A consistent focus on operations, maintenance and continuous improvement has seen the smelter easily achieve its nameplate capacity of 1.2 million tonnes per annum of copper concentrate throughput. The smelter now operates consistently at a throughput rate of about 1.38 Mtpa of copper concentrate.

Kansanshi plant concentrates are delivered by a slurry pipeline to the filter plant where the slurry is then dewatered. The concentrate is sent to the concentrate storage shed where it is blended with the concentrate from Sentinel. Concentrates from the Sentinel plant are delivered by trucks and offloaded into the concentrate storage shed. The blended concentrate is delivered to the material handling area where silica, limestone, slag and reverts are added to make up the feed to the Isasmelt furnace. Excess concentrates from Sentinel are not offloaded, but redirected to other smelters on the Copperbelt.

The Isasmelt furnace operates at a feed rate of 170 to 180 dry t/h of concentrate. Off-gases from the furnace are cooled in a vertical water tube boiler, followed by a hot electrostatic precipitator. The hot gases are delivered to the wet gas cleaning system of the acid plant where they are cooled, cleaned and mixed with cleaned off-gases from the Pierce-Smith converters (PSCs) before going to the acid plant for the production of sulphuric acid.

Matte and slag are tapped from the furnace to the electric furnace where they are allowed to separate. Slag is tapped off into ladles and tipped into slag cooling pits. The matte that is tapped is transferred by ladles to the converters.

There are four PSCs, with typically two blowing (either copper or slag blows), one on hot standby, and the fourth on turnaround. The off-gases from the PSCs are cooled and cleaned in individual gas cleaning trains dedicated to each PSC. The gas is mixed with the gas from the Isasmelt furnace and delivered to the acid plant.

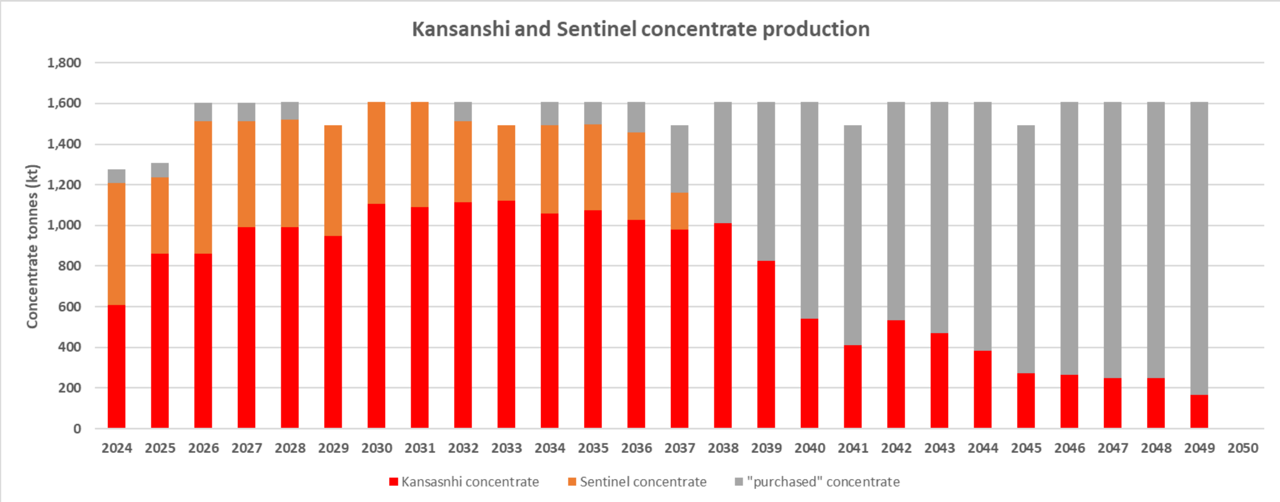
Blister from the PSCs undergoes final refining in the anode furnaces to produce anode copper. Anodes are cast using two x 18 mould-casting wheels at a rate of up to 110 t/h.

Sulphuric acid produced from the sulphur dioxide in the Isasmelt and PSC off-gases is delivered back to the Kansanshi plant where it is used for leaching of oxide ores.

17.3.1 Future concentrate treatment

Commissioning of the S3 circuit along with increases in throughput at the Sentinel plant, will result in a future concentrate production rate of up to 1.6 Mtpa (Figure 17-8). This figure shows that the smelter capacity can be filled with the addition of concentrate from other sources, including Sentinel, and which are additional to that associated with the Mineral Reserve inventory.

Figure 17-8 Concentrate production schedule



The increased concentrate production will be handled by:

- an increase in smelter throughput up to 1.6 Mtpa
- 0.2 Mtpa of concentrate to be treated through the HPL circuit
- Some concentrate marketed to other Zambian smelters, where Kansanshi and Sentinel production exceeds the smelter throughput capacity

Figure 17-9 shows the current smelter flowsheet, whilst Figure 17-10 highlights the proposed upgrades/expansions.

The smelter expansion has been partially achieved by the addition of an Isoconvert furnace, which takes the place of an additional PSC.

The existing Isasmelt furnace will be adequate for the increased throughput, but requires an upgrade to the gas handling system to treat the additional off-gas due to increased concentrate throughputs. These upgrades include increased capacity for the Induced Draft Fan, the Intermediate Blower, and the Air-Cooled Condenser in the Waste heat Recovery Boiler.

Additional oxygen will be required for the smelter and for the HPL circuit. Several options have been looked at and costed for an additional oxygen plant to cover both requirements.

Figure 17-9 Current KCS flowsheet

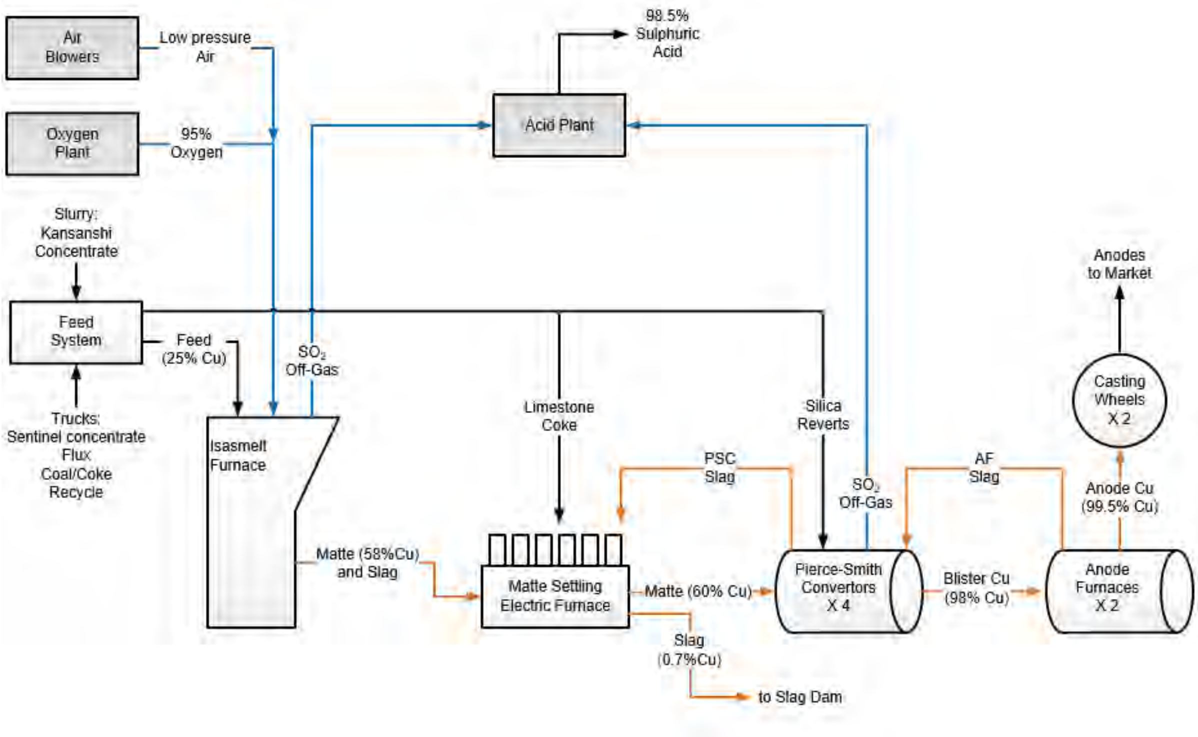
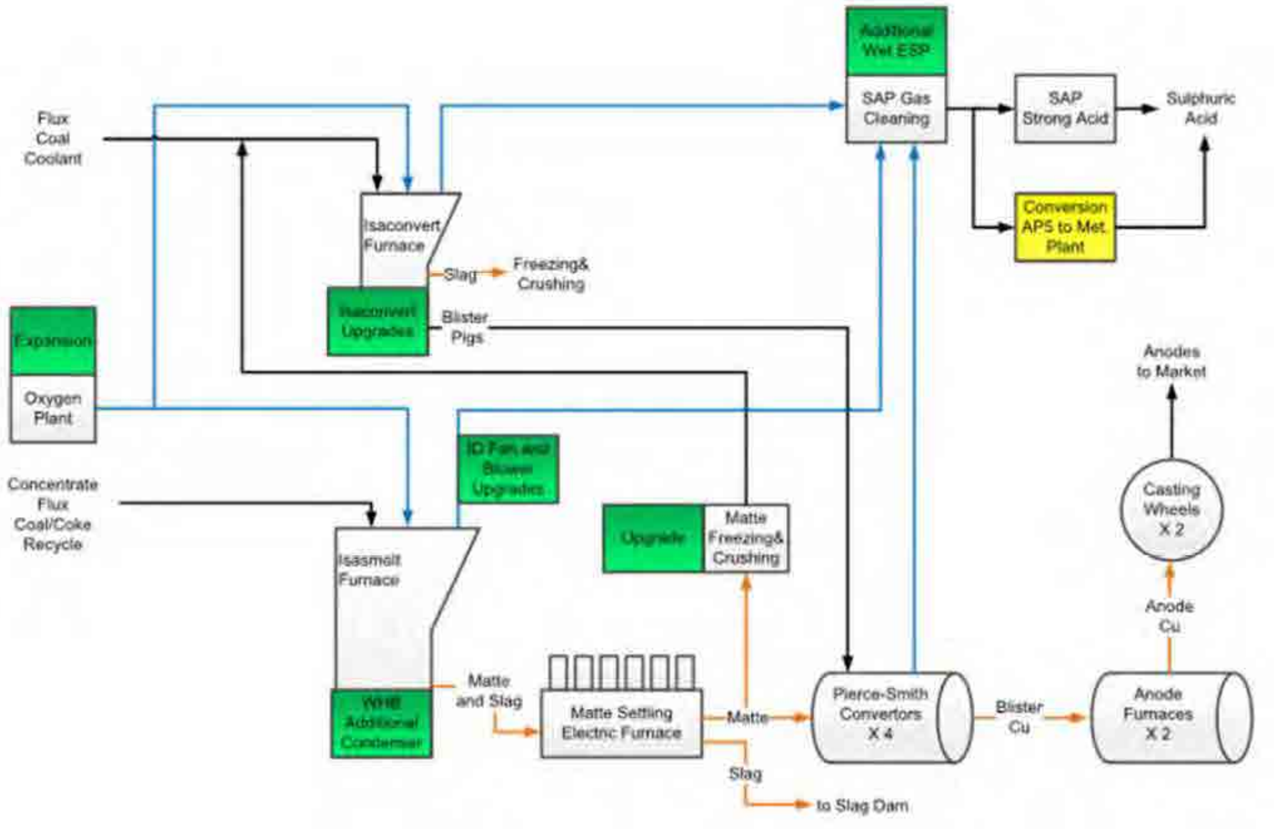


Figure 17-10 KCS flowsheet upgrades



Increased concentrate treatment will produce more off-gases containing sulphur dioxide. Additional gas handling facilities (i.e. an additional wet electrostatic precipitator) and an additional acid plant will be required. One of the mothballed sulphur burning acid plants at Kansanshi will be modified to handle this additional off-gas stream.

Concentrates from both Sentinel and Kansanshi contain significant levels of carbon (1.5 to 2.5% by weight). This carbon adds to the thermodynamic heat load of the smelter, consumes oxygen and generates additional off-gases (CO_2). Any reduction in the carbon content of the concentrates leads to an increase in the smelter throughput.

Flotation circuit modifications (addition of additional cleaner capacity and flotation columns) at both sites are aimed at reducing the carbon levels in the concentrate. Reagent trials and process optimisation are continuing to address the issues of high carbon and pyrite in flotation concentrates from both sites.

The current HPL circuit comprises two autoclaves, however the capacity of the circuit is limited by the availability of high-pressure oxygen to about 100,000 tpa of concentrate. Increased oxygen availability will increase the capacity of the circuit to approximately 200,000 tpa of concentrate before the volumetric limit of the autoclaves is reached.

Increased PLS generated from the HPL circuit can be handled in the current SX and EW circuits at Kansanshi, which are currently running below capacity.

ITEM 18 PROJECT INFRASTRUCTURE

18.1 Site layout

The Kansanshi mine has been operational since 2004, commercial production was declared in March 2005, and hence all infrastructure for the current operations is in place and well established.

18.2 Existing infrastructure

Figure 18-1 is a plan showing the location of the existing infrastructure components at the mining and processing sites. Figure 18-2 shows the processing facilities in the middle foreground, with the open pits at the top of the figure and the smelter in the lower left corner.

18.2.1 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 road was constructed to support concentrate transfer between Sentinel and the smelter during the smelter build. 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.

18.2.2 Plant buildings

The existing processing facility is serviced by a number of buildings. These include control rooms, laboratories, workshops, warehouses, security, and ablution facilities.

18.2.3 Mine services

An enlarged mining fleet services area was constructed and completed in 2015 to replace a facility impacted by the southern cutbacks of the Main Pit. Further upgrades are necessary to suit the requirements of an expanded mining fleet and to reduce the amount of maintenance work currently done in the open.

18.2.4 Waste dumps, tailings dams and pipelines

Waste Storage

Waste rock at KMP is transported to external waste storage facilities located adjacent to the working pits. In pit dumping is also planned for mined out areas of the existing pits.

Domestic waste is managed according to its ability to be reused or recycled. An onsite domestic landfill is managed by KMP.

Tailings Storage

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. Nonetheless, KMP monitors the material leaving the plant and within the tailings facilities on a daily basis.

Figure 18-1 Kansanshi Operations site plan, December 2023 (source: FQM)

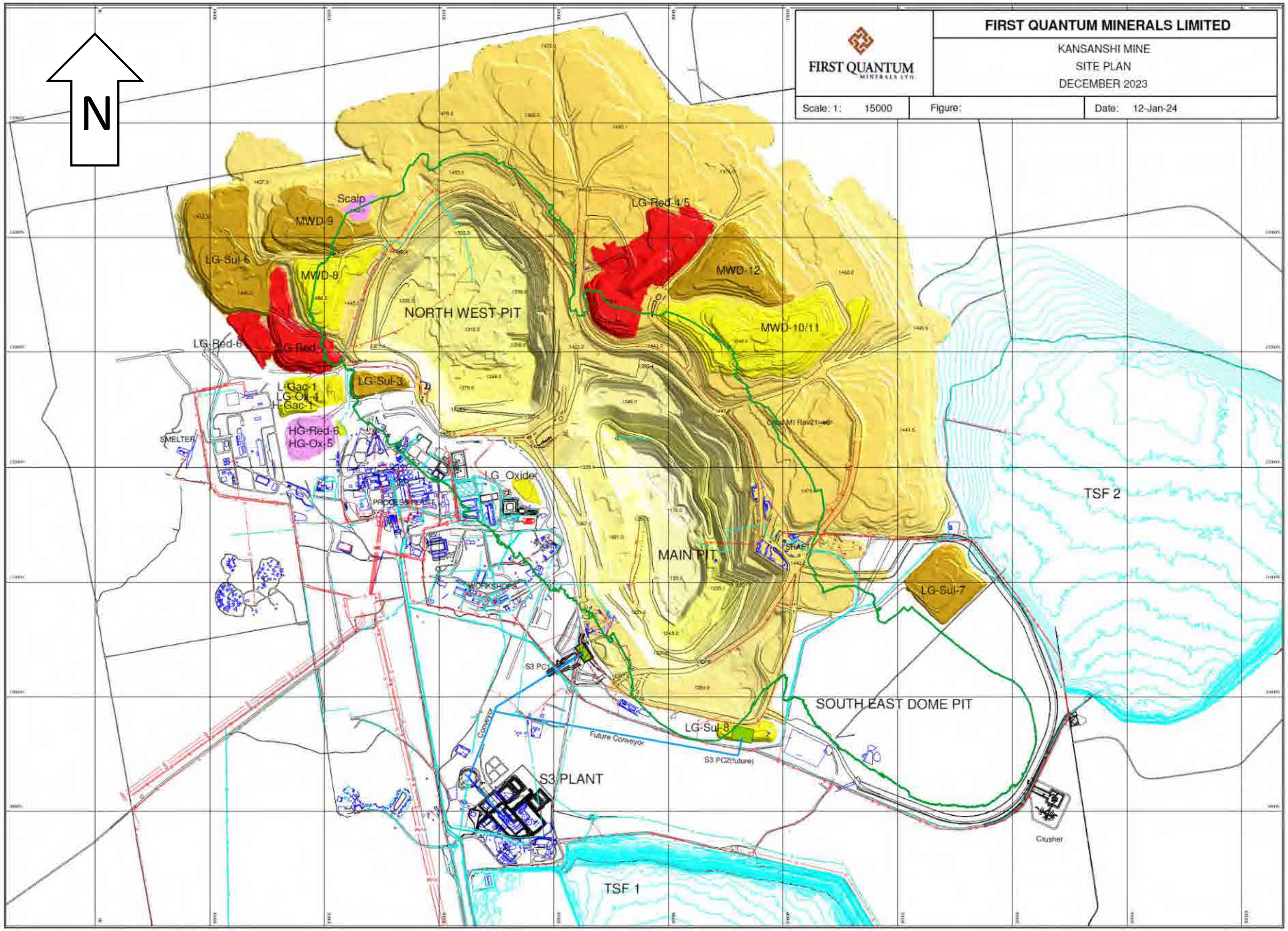


Figure 18-2 Kansanshi Operations, aerial view of the existing processing and administration facilities

Additional tailings generated from the S3 process plant will be combined with the tails from the Main Plant and transferred to TSF2. TSF2 will be converted to downstream construction methodology initially to handle the extra tails while a new TSF3 is designed and permitted.

18.2.5 Power supply

Power for the Kansanshi Operations is supplied by the Zambian national utility company (ZESCO) via two 330 kV overhead transmission lines. Power transformers at Kansanshi substation step down the utility supplied power from 330kV to 33 kV for distribution around the site. The site currently utilises 6 x 33kV feeders with an additional 3 x 33kV feeders planned at the Kansanshi substation for the below supplies:

- existing process plant (3 x feeders)
- mine pit supply (1 x feeder)
- smelter (2 x feeders)
- new S3 plant (2 x feeders)
- new South East Dome mine pit supply (1 x feeder)

The existing 33 kV mining feeder is also used to supply certain parts of the process plant, electrical mining fleet, trolley assist, dewatering, water harvesting, decant and other ancillary support services. These loads are either supplied directly from the 33 kV supply or transformed down to 6.6 kV or 525 V.

A number of 33 kV substations exist within the smelter and processing plant, supplying 3 MVA and 2 MVA transformers (one per motor control centre) which in turn step down to 525V, this being the main low voltage used for motors and other equipment.

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. This plant will be available until 2037 when it will make way for a pit extension.

The S3 Project will include a 6MW standby diesel plant connected to the S3 33 kV substation and will provide backup to the critical loads on the S3 plant.

18.2.6 Water supply and water systems

Process water demand is currently 91.4 ML/day and will increase to 149.8 ML/day when S3 is fully operational.

Process water is sourced primarily from mine dewatering activities and supplemented with decant water from the tailings storage facilities as well as pumping from the fresh water dam and from the Solwezi River.

Potable water is pumped to site from the borefield.

18.2.7 Air supply systems

A range of air supply systems have been installed 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
- high pressure concentrate filtration air – provided for the concentrate pressure filters
- flotation air - low pressure air supplied to the flotation circuits

- blower air – utilised in the smelter for primary smelting and copper converting

18.2.8 Fire protection systems

Fire detection, alarms, and suppression systems are installed in all substation buildings, including the main 33 kV substation building. Fire detection facilities have been installed where appropriate, for example for the SX trains. All plant areas have fire hydrants and hand held extinguishers installed in strategic locations.

18.3 S3 Expansion Project status

As noted in the 2020 Technical Report (FQM, September 2020), construction of the S3 expansion plant was halted in late 2013. At that point in time, much of the civil and structural work on the plant site had been completed including extensive piling and construction of the stockpile tunnel, mill structure and concrete foundations for most of the major equipment.

A fresh water dam with a capacity of 3.3 million m^3 had also been completed and is now supplying water to the current operations.

A decision was made to recommence work on the S3 plant in 2022. Detailed engineering (i.e., structural, mechanical and piping) is complete with electrical, instrumentation and control designs expected to close out by the end of Q3 2024. Specific progress has been as follows:

- Earthworks and structural concrete for major structures are completed and the trades are now focused on piping corridors and servitudes, followed by the balance of plant infrastructure.
- The primary crusher pocket excavation is completed and the site construction team is in the process of building the gabion wall. The primary crusher super structure steel has arrived on site and is currently being assembled. The conveyor transfer stations pads and concrete are under construction, in readiness for structural steel arrival in late Q2 2024.
- The crushed ore stockpile civil and earthworks have been completed, whilst the reclaim tunnel and equipment are expected on site in Q2 of 2024 for installation.
- The pebble crusher facility structural steel is currently under construction and the crusher is expected on site in Q3 of 2024.
- Mill building steel and concrete modifications have been completed and mill assembly is due to commence once the last of the mill components arrive on site in Q2 of 2024.
- Flotation pedestals and concrete works have been completed and ready for receipt of tanks and cells in late Q2.

Figure 18-3 Selection of S3 progress photos

The S3 Project commissioning is on schedule for Q2 2025 with ramp up commencing in Q3 2025.

The mining fleet has been procured and deliveries commenced in the second half of 2023 with the first new trucks and shovels already deployed in the SED pit.

18.4 Smelter expansion status

Engineering has been completed on the related smelter expansion and orders have been placed for key long-lead items associated with the oxygen plant, acid plant, and wet electrostatic precipitators.

18.4.1 Tie-in work

All major equipment tie-ins were completed during the 2021 major smelter shut to facilitate integration of the expansion equipment with little to no downtime on the smelter operations.

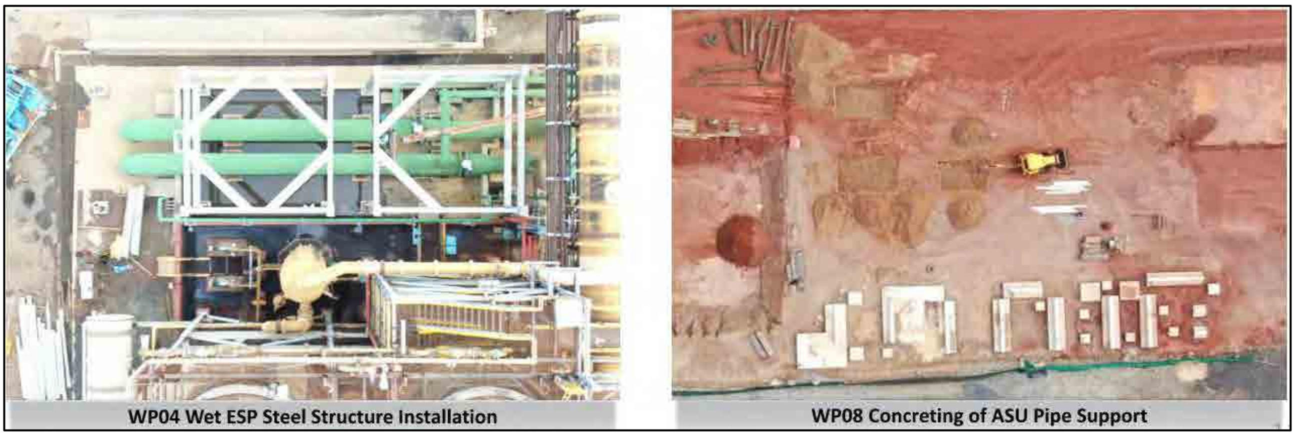
18.4.2 Current site status

- All major earth works have been completed and civil works for major equipment is in progress.
- The waste heat boiler condenser is on site, partially assembled and ready to be lifted into position in Q2 2024.
- The wet electrostatic precipitator (WESP) steel structure construction is in progress with the casings to be installed in Q2 2024 (Figure 18-4.
- Fibre reinforced ducting from the smelter to acid plant (AP) 5 is currently being installed.
- Manufacturing of all oxygen plant equipment is to be completed in Q2 2024 with installation to commence in Q3 2024.
- AP 5 modifications are in progress with major equipment all being shipped by Q2 2024.
- Concreting of the air supply unit pipe supports is in progress (Figure 18-4).

18.4.3 Shut down work

Intermediate blower and induced draft fan upgrades are to be implemented during the smelter shut in June 2025.

Figure 18-4 Smelter progress photos



ITEM 19 MARKET STUDIES AND CONTRACTS

With the advent of on-site smelting at KMP, the supply and sale of concentrate to local smelters essentially ended in 2015. Anode product from the smelter and cathode product from the SXEW plant are now the main saleable products.

Currently, all anode and cathode product is being sold through the Company's internal marketing division, FQM Trading.

In November 2023, a ten-year power supply agreement (backdated to January 2023) was signed between the Company and ZESCO, the Zambian state energy provider. Within the agreement term, ZESCO is committed to supplying renewably sourced power to the Project.

In response to a drought aggravated by El Niño, ZESCO has revealed a plan to reduce power supply to the Zambian mining sector over the period from May 1, 2024 to December 31, 2024. The Company anticipates that it will be able to substitute the power curtailed by ZESCO with power sourced from elsewhere in southern Africa, thereby avoiding any major interruptions to its operations during this period.

In view of the property being an established operation, no further studies, analyses or QP review have been undertaken in respect of product marketing, commodity price projections, product valuations, market entry strategies or product specification requirements.

ITEM 20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental setting

The current processing facilities are located immediately south of the open pits. The 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 S3 sulphide plant, immediately north of the existing tailings dams, has been selected based on availability of suitable space, proximity to existing infrastructure and roads, is outside the future mining envelope, and takes consideration of geotechnical conditions. The site is already disturbed from previous construction activities, and is situated in degraded woodland surroundings.

The remaining vegetation at the S3 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.

The Operations site is drained by the Solwezi River to the west and the Kifubwa River to the south. The rivers join to the south of the Surface Rights Boundary area. A large wetland (dambo) lies between the mine and TSF1 and drains south east into the Kifubwa River.

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.

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

- TSF2 – the design and construction of the new sulphide plant tailings dam facility
- Smelter – the design and construction of the 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.

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 compliance with statutory Zambian effluent and emissions limits. KMP currently tracks several hundred commitments associated with the more recent EIAs.

These commitments have been incorporated into a Consolidated Environmental Management Plan (EMP), approved by ZEMA. The commitments are subject to regular audits by the Zambian Mines Safety Department (MSD) and by ZEMA.

An Environmental Management System (EMS) is in place to improve overall performance and improve adherence to both the policy and legal requirements. The EMS accords with the ISO 14001:2015 standard and is regularly audited by external parties.

20.3.1 Tailings management

In accordance with Zambian statutory requirements and international best practice, quarterly Dam Safety Inspections are undertaken at both tailings dams by an experienced independent tailings dam practitioner. An annual Independent Technical Review is provided by a recognised global tailings dam expert.

In addition to the inspections, regular reviews of tailings dam stability are completed on site and by independent, industry leading experts. Each dam has a facility specific Trigger Action Response Plan (TARP) which is used to alert to various levels of abnormal behaviour from daily, weekly and other inspections and from instrumentation readings. The TARP is intended to trigger specific mitigating actions, with evacuate being the final action.

The Company has stated its ambition of aligning the KMP operations with the performance aspects of the Global Industry Standard on Tailings Management (GISTM). Current initiatives include enhanced site oversight, improved TSF instrumentation, monitoring and behaviour analyses, in addition to improved Learning and Development programmes.

Monitoring of tailings dam stability and environmental performance will continue post closure to ensure that adequate factors of safety are maintained.

20.4 Resettlement

Some of the recent developments at KMP have extended the site boundaries and local residents have been resettled. 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.

Numerous resettlement programmes around the KMP site have been successfully completed in recent years.

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, and grievances are investigated and resolved within a time period acceptable to each party.

20.6 Environmental monitoring

KMP has established a comprehensive environmental monitoring network within and immediately adjacent to its operational footprint. This network covers a range of receiving media including but not limited to surface and groundwater, and air quality. Fixed monitoring points have been established at all surface water discharge points in accordance with Zambian legislation and specific licence conditions. All compliance related surface water samples are analysed at independently accredited off site laboratories, while internal monitoring for operational control and early warning systems includes a combination of real time probes as well as on site analysis.

There exists a comprehensive groundwater monitoring network around the tailings storage facilities and the mine waste rock dumps. The positioning of groundwater monitoring points around mining infrastructure has been informed by industry standard tools such as modelling and geotechnical surveys. A number of monitoring points in the communities surrounding the mining operation provide an early warning of potential issues. Three air quality monitoring stations and emissions monitoring probes provide real time air quality data from the smelter.

KMP maintains a working water balance model and monitors the volumes of surface water withdrawn and discharged. In recent years several initiatives have been launched to improve water reuse and to reduce water withdrawal requirements.

Monitoring of all infrastructure including the tailings dams and waste rock dumps will continue post closure and will be guided by ongoing monitoring results and standard modelling and predictive tools.

20.7 Mine closure

The main environmental liabilities at Kansanshi will arise at closure and are related to the decommissioning and dismantling of the process plants and ancillary infrastructure, and the rehabilitation of the tailings dams, open pit mine and waste rock dumps.

The Kansanshi Mine asset retirement obligation (ARO) is reviewed annually. Where possible, to reduce final closure liabilities, rehabilitation activities are advanced concurrently with ongoing operations.

At the end of December 2023, the ARO at Kansanshi was estimated to be $128.2M[8]. In accordance with National Legislation, Kansanshi contributes to an Environmental Protection Fund administered by the Zambian Mines Safety Department.

[8] The unscheduled closure cost estimate, as at the end of December 2023, is $103.8M.

ITEM 21 CAPITAL AND OPERATING COSTS

Information provided under this item relates specifically to the capital and operating costs compiled by FQM staff initially for the purposes of five-year budgeting and forecasting, and subsequently for development of a cashflow model to support the Mineral Reserve estimate (Item 15).

Arising from this compilation process was a review of operating cost inputs for the purposes of pit optimisation, followed by the projection of operating costs for inclusion in the cashflow model. In respect of the mining and process operating cost projections, this review was undertaken by Michael Lawlor (QP) and Andrew Briggs (QP), respectively, each of FQM.

21.1 Capital costs

The capital cost provisions included in the Kansanshi cashflow model as tabulated in Item 22, total $1,688 M from January 2024, or $1,942 M inclusive of amounts spent in the preceding two years (Table 21-1).

The three largest items, i.e. the S3 expansion, the additional mining fleet and the South East Dome pre-strip, are projected to cost a total of $1.25 B inclusive of amounts spent in the preceding two years.

Table 21-1 Kansanshi capital cost provisions

	TOTAL	BEFORE 2024	FROM 2024	2024	2025	2026	2027	2028	2029	2030
Mining capital ($M)										
Mining fleet	**$241.3**	**$63.0**	**$178.3**	$178.3						
subtotal ($M)	**$241.3**	**$63.0**	**$178.3**	**$178.3**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**
Processing capital ($M)										
S3 expansion	**$775.0**	**$151.1**	**$623.9**	$429.5	$152.5	$41.8				
Smelter expansion / upgrade	**$115.0**	**$40.0**	**$75.0**	$47.0	$28.0					
Process plant upgrade	**$13.2**	**$0.0**	**$13.2**	$3.9	$8.7	$0.6				
TSF expansion	**$116.0**	**$0.0**	**$116.0**	$23.7	$17.9	$38.5	$23.9	$11.8		
S3 site infrastructure and ancillary equipment	**$27.8**	**$0.0**	**$27.8**	$9.8	$6.5	$3.9	$4.0	$3.5		
TSF 3	**$94.0**	**$0.0**	**$94.0**					$25.0	$69.0	
subtotal ($M)	**$1,140.9**	**$191.1**	**$949.8**	**$513.9**	**$213.7**	**$84.9**	**$27.9**	**$40.3**	**$69.0**	**$0.0**
Other capital ($M)										
SE Dome pre-strip	**$236.0**	**$0.0**	**$236.0**	$68.6	$121.5	$45.9				
Other SE Dome stripping	**$182.9**	**$0.0**	**$182.9**			$61.5	$79.0	$42.4		
Dewatering shaft & associated development	**$21.6**	**$0.0**	**$21.6**	$21.6						
Trolley line Installations	**$30.7**	**$0.0**	**$30.7**	$11.6	$10.9	$3.8	$4.3			
Second primary crusher	**$70.0**	**$0.0**	**$70.0**		$5.0	$20.0	$45.0			
Gravity gold	**$0.7**	**$0.3**	**$0.3**	$0.3						
Other (aggregate crusher, HPL oxygen upgrade, and other)	**$18.2**	**$0.0**	**$18.2**	$17.5	$1.0	-$0.2	-$0.1	$0.0		
subtotal ($M)	**$560.0**	**$0.3**	**$559.6**	**$119.7**	**$138.4**	**$131.1**	**$128.2**	**$42.3**	**$0.0**	**$0.0**
Total ($M)	**$1,942.1**	**$254.4**	**$1,687.7**	**$811.9**	**$352.0**	**$216.0**	**$156.2**	**$82.7**	**$69.0**	**$0.0**

In relation to the $241 M of mining capital provisions, Table 21-2 itemises the amounts for the purchase of production drills, primary shovels, haul trucks and ancillary equipment. Where listed, the capital items also provide for delivery, assembly, commissioning and truck tyres. Timing differences reflective of vendor payment arrangements are evident between Table 21-1 and Table 21-2, in respect of the annual mining fleet costs.

In relation to the $775 M listed in Table 21-1 for the S3 expansion capital:

- $480 M for direct costs inclusive of mechanical and electrical items, steel and platework, concrete, piping, earthworks and plant buildings
- $295 M for indirect costs inclusive of engineering charges, spares and first fills, freight charges, support and construction costs, commissioning and contingency/escalation (28%), plus taxes

In relation to the smelter upgrade cost listed in Table 21-1, the figure of $115 M covers direct costs inclusive of:

- $16.3 M for acid plant 5
- $26 M for the oxygen plant

- $7.4 M for the matte granulator
- $6.2 M for wet electrostatic precipitators (WESP)
- $53 M for smelter plant-wide services
- $6.3 M for miscellaneous facilities

Table 21-2 Capital cost provisions for the new mining fleet

	QTY	TOTAL	2022	2023	2024	2025
Drills						
Epiroc Pit Viper PV231E Drills 7-5/8"x 254mm	6	**$26.4**	$4.0	$0.0	$22.4	$0.0
Delivery, assembly, commissioning		**$2.8**	$0.4	$0.0	$2.1	$0.3
subtotal		**$29.2**	**$4.4**	**$0.0**	**$24.5**	**$0.3**
Epiroc T45-10 Long Mast Drills	8	**$3.7**	$0.6	$3.2	$0.0	$0.0
Delivery, assembly, commissioning		**$0.2**	$0.0	$0.2	$0.0	$0.0
subtotal		**$4.0**	**$0.6**	**$3.4**	**$0.0**	**$0.0**
Drills subtotal	**14**	**$33.1**	**$5.0**	**$3.4**	**$24.5**	**$0.3**
Shovels						
Hitachi EX5600-7E Shovel 31m3 4 pass = 220t	6	**$35.9**	$1.8	$0.0	$34.1	$0.0
Delivery, assembly, commissioning		**$6.7**	$0.0	$0.0	$6.7	$0.0
Shovels subtotal	**6**	**$42.6**	**$1.8**	**$0.0**	**$40.8**	**$0.0**
Trucks						
Hitachi EH4000-AC3 Dump Trucks c/w Trolley & Tray	36	**$100.0**	$5.0	$25.4	$69.7	$0.0
Delivery, tyres, assembly, commissioning		**$17.3**	$0.0	$3.5	$13.7	$0.0
Trucks subtotal	**36**	**$117.3**	**$5.0**	**$28.9**	**$83.4**	**$0.0**
Dozers						
Komatsu D375 Dozers	6	**$6.0**	$1.2	$4.8	$0.0	$0.0
Delivery, assembly, commissioning		**$0.0**	$0.0	$0.0	$0.0	$0.0
subtotal		**$6.0**	**$1.2**	**$4.8**	**$0.0**	**$0.0**
Komatsu D475 Dozers	2	**$3.1**	$0.6	$2.5	$0.0	$0.0
Delivery, assembly, commissioning		**$0.0**	$0.0	$0.0	$0.0	$0.0
subtotal		**$3.1**	**$0.6**	**$2.5**	**$0.0**	**$0.0**
Dozers subtotal	**8**	**$9.1**	**$1.8**	**$7.3**	**$0.0**	**$0.0**
Graders						
Komatsu GD825 Graders	6	**$3.6**	$0.7	$2.5	$0.4	$0.0
Delivery, assembly, commissioning		**$0.0**	$0.0	$0.0	$0.0	$0.0
Graders subtotal	**6**	**$3.6**	**$0.7**	**$2.5**	**$0.4**	**$0.0**
Other						
Komatsu WA600-6R Cable Handlers	3	**$3.0**	$0.6	$2.3	$0.0	$0.0
Delivery, assembly, commissioning		**$0.0**	$0.0	$0.0	$0.0	$0.0
subtotal		**$3.0**	**$0.6**	**$2.3**	**$0.0**	**$0.0**
Komatsu WA500-6R Stemming Loader c/w Bucket	2	**$1.0**	$0.2	$0.8	$0.0	$0.0
Delivery, assembly, commissioning		**$0.0**	$0.0	$0.0	$0.0	$0.0
subtotal		**$1.0**	**$0.2**	**$0.8**	**$0.0**	**$0.0**
Other subtotal	**5**	**$4.0**	**$0.8**	**$3.1**	**$0.1**	**$0.0**
Miscellaneous equipment						
Misc. equipment subtotal		**$24.0**	**$0.0**	**$5.0**	**$18.9**	**$0.0**
Mining equipment grand total	**75**	**$233.7**	**$15.1**	**$50.2**	**$168.1**	**$0.3**
PO incurred forecast		**$241.1**	$15.1	$55.8	$169.9	$0.3
Cashflow model					**$178.3**	

21.1.1 Comparison with previous capital cost estimates

The 2020 Technical Report listed a provision of $408 M for the S3 expansion, including concentrate production and mixed ore treatment upgrades. This provision now stands at $775 M, with the increase attributable to scope and design changes, inclusion of additional areas and some inflation related costs. The primary scope changes relate to the introduction of CPS into rougher flotation, combined with cleaning and tailings pumping facilities integrated with the existing sulphide (S2) facility. In addition, the gravity gold circuits have been expanded and improved pebble crushing capabilities have been introduced. The S3 budget was also increased to reflect current market prices following the COVID pandemic.

The 2020 Technical report also listed a provision of $80 M for the smelter upgrade, and that provision has now increased to $115 M. The 2020 provision was estimated prior to completion of an internal engineering study, arising from which was a number of scope changes and pricing updates reflective of then current market changes.

21.1.2 Sustaining capital costs

The sustaining cost provisions included in the Kansanshi cashflow model, as tabulated in Item 22, total $2,319.6 M (Table 21-3). These provisions cover mining fleet replacements, mining fleet component change-outs, in addition to process plant and smelter sustaining allowances.

21.1.3 Mine closure provisions

Mine closure provisions (also referred to as asset retirement obligations, or AROs) are routinely reviewed and updated by external consultants at the end of each calendar year. This review involves a thorough process of itemising closure and post-closure cost estimates for items such as infrastructure decommissioning, site rehabilitation, social aspects of closure, and monitoring of groundwater and rehabilitated areas. The estimates for these comprehensively listed activities are typically built-up from benchmarked unit costs being applied to the areas utilised by mining, processing, and related activities during the life of mine.

A summary of the itemised estimate as at the 31st December 2023 is provided in Table 21-4.

Table 21-3 Kansanshi sustaining cost provisions

	TOTAL	FROM 2024	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
MINING															
Mining fleet replacements	**$864.0**	**$864.0**	$35.0	$107.4	$141.6	$76.5	$29.7						$94.8	$94.8	$94.8
Mining fleet component change out	**$686.4**	**$686.4**	$31.8	$36.4	$24.8	$20.6	$24.1	$40.3	$40.2	$40.7	$40.7	$40.6	$40.7	$40.7	$40.7
Ancilliary fleet replacements	**$36.0**	**$36.0**						$2.0	$2.0	$2.0	$2.0	$2.0	$2.0	$2.0	$2.0
Hitachi shortfall make-up	**$83.6**	**$83.6**				$41.8	$41.8								
Geotech+Hydrology	**$4.4**	**$4.4**	$1.8	$1.7	$0.5		$0.5								
Other mining projects	**$19.8**	**$19.8**	$8.5	$6.0	$1.9	$2.8	$0.7								
Subtotal ($M)	**$1,694.2**	**$1,694.2**	**$77.0**	**$151.4**	**$168.7**	**$141.7**	**$96.8**	**$42.3**	**$42.2**	**$42.7**	**$42.7**	**$42.6**	**$137.4**	**$137.4**	**$137.4**
PROCESSING															
Processing LT Provision	**$240.0**	**$240.0**						$16.1	$16.1	$16.1	$16.8	$16.8	$16.8	$16.8	$16.8
Other Processing Projects	**$8.3**	**$8.3**	$2.9	$2.2	$0.9	$1.2	$1.1								
Subtotal ($M)	**$248.3**	**$248.3**	**$2.9**	**$2.2**	**$0.9**	**$1.2**	**$1.1**	**$16.1**	**$16.1**	**$16.1**	**$16.8**	**$16.8**	**$16.8**	**$16.8**	**$16.8**
SMELTING															
Smelter ongoing Sustaining	**$50.7**	**$50.7**	$1.9	$2.2	$1.8	$2.5	$0.9	$2.3	$2.3	$2.3	$2.3	$2.3	$2.3	$2.3	$2.3
Isasmelt Shutdown (Based On The 2017 Shutdown Actuals)	**$180.0**	**$180.0**	$7.2	$22.8				$30.0				$30.0			
Subtotal ($M)	**$230.7**	**$230.7**	**$9.1**	**$25.0**	**$1.8**	**$2.5**	**$0.9**	**$32.3**	**$2.3**	**$2.3**	**$2.3**	**$32.3**	**$2.3**	**$2.3**	**$2.3**
INRASTRUCTURE AND OTHER															
Infrastructure S3 Related	**$9.8**	**$9.8**	$9.8												
TSF	**$53.0**	**$53.0**	$41.3	$1.7	$1.3	$0.5		$2.1	$0.1	$0.1	$0.5		$2.1	$0.1	$0.1
Other Engineering Projects	**$17.0**	**$17.0**	$8.3	$7.2	$0.4	$1.1									
Other site projects	**$66.6**	**$66.6**	$7.0	$13.1	$16.1	$14.4	$16.0								
Subtotal ($M)	**$146.4**	**$146.4**	**$66.3**	**$22.1**	**$17.8**	**$16.0**	**$16.0**	**$2.1**	**$0.1**	**$0.1**	**$0.5**	**$0.0**	**$2.1**	**$0.1**	**$0.1**
Grand Total ($M)	**$2,319.6**	**$2,319.6**	**$155.3**	**$200.8**	**$189.2**	**$161.4**	**$114.7**	**$92.7**	**$60.7**	**$61.1**	**$62.3**	**$91.7**	**$158.6**	**$156.6**	**$156.6**

	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050	2051
MINING															
Mining fleet replacements	$94.8	$94.8													
Mining fleet component change out	$40.1	$38.0	$30.0	$35.7	$24.1	$22.3	$16.7	$10.7	$3.7	$3.0					
Ancilliary fleet replacements	$2.0	$2.0	$2.0	$2.0	$2.0	$2.0	$2.0	$2.0	$2.0	$2.0					
Hitachi shortfall make-up															
Geotech+Hydrology															
Other mining projects															
Subtotal ($M)	**$136.8**	**$134.8**	**$32.0**	**$37.7**	**$26.1**	**$24.3**	**$18.7**	**$12.7**	**$5.7**	**$5.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**
PROCESSING															
Processing LT Provision	$16.8	$16.7	$16.7	$8.2	$8.2	$8.2	$8.2	$8.2	$8.2	$8.2					
Other Processing Projects															
Subtotal ($M)	**$16.8**	**$16.7**	**$16.7**	**$8.2**	**$8.2**	**$8.2**	**$8.2**	**$8.2**	**$8.2**	**$8.2**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**
SMELTING															
Smelter ongoing Sustaining	$2.3	$2.3	$2.3	$2.3	$2.3	$2.3	$2.3	$2.3	$2.3	$2.3					
Isasmelt Shutdown (Based On The 2017 Shutdown Actuals)	$30.0				$30.0				$30.0						
Subtotal ($M)	**$32.3**	**$2.3**	**$2.3**	**$2.3**	**$32.3**	**$2.3**	**$2.3**	**$2.3**	**$32.3**	**$2.3**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**
INRASTRUCTURE AND OTHER															
Infrastructure S3 Related															
TSF	$0.5		$2.1	$0.1	$0.1	$0.5		$0.1	$0.0	$0.0					
Other Engineering Projects															
Other site projects															
Subtotal ($M)	**$0.5**	**$0.0**	**$2.1**	**$0.1**	**$0.1**	**$0.5**	**$0.0**	**$0.1**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**
Grand Total ($M)	**$186.4**	**$153.8**	**$53.0**	**$48.4**	**$66.7**	**$35.3**	**$29.2**	**$23.4**	**$46.2**	**$15.5**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**

Table 21-4 Kansanshi closure cost provisions, as at December 2023

		($M)
Closure components		
	Dismantling of plant, buildings and related structures	$26.5
	Demolition of steel buildings and structures	$0.0
	Demolition of reinforced concrete structures buildings	$0.1
	Rehabilitation of roads and paved surfaces	$2.2
	Demolition of offices, workshops and residential buildings	$9.9
	Demolition and rehabilitation of railway lines	$0.0
	Fencing	$0.1
	Disposal of other linear infrastructure	$1.7
	Disposal and handling of waste	$4.9
	Making good of infrastructure	$0.0
	Subtotal	**$45.5**
	Mining areas	
	Sealing of shafts, adits and inclines	$0.4
	Open pit rehabilitation including final voids and ramps	$5.6
	Rehabilitation of overburden, processing residues, spoils and waste rock dumps	$1.7
	Rehabilitation of processing waste deposits and evaporation ponds (polluting potential)	$27.3
	Rehabilitation of subsided areas	$0.0
	Subtotal	**$35.1**
	Surface rehabilitation	
	General surface rehabilitation	$8.4
	Subtotal	**$8.4**
	Surface runoff measures	
	Dambo wetlands - reinstatement of aquatic health	$1.8
	Subtotal	**$1.8**
	Pre-site relinquishment aspects	
	Initial monitoring and aftercare	$21.4
	Subtotal	**$21.4**
	P&Gs, contingencies and additional allowances	
	Preliminary and general	$13.6
	Contingencies	$0.0
	Closure related aspects	$2.5
	Subtotal	**$16.1**
	Residual and latent aspects	
	Residual aspects	$0.0
	Latent aspects	$0.0
	Contingencies on residual and latent aspects	$0.0
	Subtotal	**$0.0**
	Total scheduled closure costs	**$128.2**

The total cost provision as at December 2023 is included in the Mineral Reserve cashflow model.

21.2 Operating costs

21.2.1 Mining costs

In relation to mine operating cost projections, estimates were completed through a series of steps, essentially as follows:

Base Cost Assumptions

1. Base costs (variable plus fixed) were adopted from the current five year business plan period (to 2028), reflecting:
   a) fuel costs ranging from $1.01/L to $0.91/L
   b) power unit cost of 9.57c/kWh
   c) drill/blast costs related to current free-dig and blasted rock proportions ranging from $1.57/bcm to $1.85/bcm
   d) current excavating/loading costs trending towards costs relative to higher utilisation face shovels plus smaller hydraulic excavators and loaders operating at lower utilisation

e) the use of medium scale capacity trucks (i.e. 180 t to 220 t capacity), hauling ore to the existing ROM pad and to the S3 crushers, and hauling deeper waste from high strip ratio cutback phases
f) haulage along defined ore and waste routes applicable to the current budget plan, with an explicit identification of routes for which extended trolley assisted haulage could be adopted
g) average load and haul cost for the business plan period ranging from \$2.39/bcm to \$2.86/bcm

LOM Cost Assumptions

2. LOM costs beyond 2028 are based on extrapolated 5 year business plan base costs reflecting:

a) the impact of forecasted operational improvements
b) an increased fuel price, from a longer term consensus view, of \$1.05/L
c) a longer term power cost of 9.57c/kWh
d) projection of the same drill/blast unit costs as at 2028, with the exception of fuel and consumable variable costs, each of which were adjusted to account for changing free-dig proportions and longer term prices
e) projection of the same load/haul unit costs as at 2028, with the exception of fuel, which changes according to the increased price
f) LOM mine haulage simulation considering variable haulage profiles (distance, mining elevation, haulage gradient, trolley assist and destination) to inform haulage fleet diesel consumption
g) projection of the same ancillary equipment and other miscellaneous mining costs as at 2028

Operational improvements

Reflecting on past operational performance and current budget practices, opportunities have been identified to reduce the base mining costs through operational improvements across several operational mining areas. Given the magnitude of hauling costs (~40% of mining costs), the primary driver to reduce mining unit costs is to reduce the size of the required truck fleet compared to the base through improved haul truck productivity.

To this end, the most impactful operational improvement areas in regards to mining cost savings have been identified as follows:

1. Trolley assist installations

a) Increasing the proportion of the haulage cycle on trolley assist to increase total speed, productivity, and hence overall capacity of the haulage fleet. Trolley assist is most impactful when installed on ramps but is also economic when installed at moderate and lower gradients;
b) The quantifyable benefits of trolley assist are:

i) Double the travel speed of loaded trucks on +10% ramps from ~11kmph to ~22kmph (a saving in cycle time of > 2 mins for each kilometre travelled under trolley assist) resulting in increased truck productivity;
ii) A ~90% reduction in diesel consumption compared to loaded on-ramp haulage without trolley assist;
iii) Historical analysis indicates realised diesel cost saving of \$0.17/t/km under trolley assist although it is marginally offset by an increase in electricity cost of \$0.04/t/km.

2. Truck productive time

   a) Forecasted implemention of additional short interval control measures and improved truck fleet dispatch practices will increase the amount of time haul trucks spend performing their primary productive function (i.e. increased tyre turn time);

   b) Targeted reduction of non-productive time (ie. queueing at source and sink) will result in an overall truck fleet productivity improvement of 4.5% and a proportional impact on overall haulage fleet capacity requirements.

3. Equipment availability and reliability

   a) Expanding and continuing the implementation of the CARE (mobile equipment maintenance) programme to increase the competency of the artisan workforce will improve maintenance practices and capability. The result is a forecasted decrease in unplanned maintenance occurrences and a 1% increase in fleet availability compared to the base yielding a 2% increase in fleet productive time and a resulting reduction in haulage fleet requirements.

The combined impact of these mining fleet related operational improvements translate to a 9% reduction in overall haulage fleet requirements to achieve the same LOM production targets.

4. Drill and blast optimisation

   a) Optimisating of drill patterns for both ore and waste through the introduction of 127mm drill holes as well as improving specialty blasting applications and the sourcing of alternative and more cost effective consumables translates to a 10% drill and blast unit cost reduction over the LOM timeframe.

Adjustment of the base mining costs (variable plus fixed) to reflect the above described operational improvements are listed in Table 21-5.

Table 21-5 Adjustments to base mining costs to reflect operational improvements

Mining Costs	Adjustments (av. from 2029)	
	$/bcm	%
Employee costs	-$0.027	-3.1%
Contractor costs	-$0.031	-6.5%
Fuel costs	-$0.240	-12.5%
Consumable costs	-$0.285	-11.2%
Electricity costs	$0.003	23.0%
Maintenance costs	-$0.004	-0.7%
Other mining costs	-$0.014	-7.5%
Total	-$0.598	-9.0%

Summary of unit mining cost estimates

As a final step, the adjusted and projected base mining costs were further refined to account for foreign exchange based components and driven by local currency fluctuations.

For the purposes of the cashflow model described in Item 22, the estimated unit costs that apply for each year by activity and cost centre, are listed in Table 21-6.

Table 21-6 Mining costs for cashflow modelling

Year	Mining costs by activity					Mining costs by cost centre							
	Drill & Blast $/bcm	Load & Haul $/bcm	Ancillary Eqt $/bcm	Mining Other $/bcm	TOTAL $/bcm	Employees $/bcm	Contractors $/bcm	Fuel $/bcm	Consumables $/bcm	Electricity $/bcm	Maintenance $/bcm	Other $/bcm	TOTAL $/bcm
2024	$1.67	$2.86	$1.15	$1.98	**$7.67**	$0.61	$0.79	$2.03	$2.86	$0.06	$1.23	$0.10	**$7.67**
2025	$1.85	$2.40	$0.87	$1.28	**$6.39**	$0.47	$0.59	$1.68	$3.02	$0.04	$0.51	$0.09	**$6.39**
2026	$1.59	$2.44	$0.79	$1.28	**$6.11**	$0.47	$0.51	$1.74	$2.69	$0.04	$0.57	$0.09	**$6.11**
2027	$1.57	$2.50	$0.62	$1.20	**$5.89**	$0.42	$0.48	$1.70	$2.59	$0.04	$0.58	$0.08	**$5.89**
2028	$1.70	$2.39	$0.51	$1.18	**$5.78**	$0.40	$0.24	$1.69	$2.80	$0.05	$0.53	$0.07	**$5.78**
2029	$1.52	$2.41	$0.51	$1.22	**$5.66**	$0.40	$0.24	$1.72	$2.62	$0.05	$0.54	$0.08	**$5.66**
2030	$1.49	$2.54	$0.53	$1.26	**$5.81**	$0.42	$0.25	$1.88	$2.60	$0.05	$0.55	$0.08	**$5.81**
2031	$1.48	$2.55	$0.52	$1.25	**$5.80**	$0.41	$0.25	$1.90	$2.58	$0.05	$0.54	$0.08	**$5.80**
2032	$1.38	$2.56	$0.52	$1.25	**$5.71**	$0.41	$0.24	$1.92	$2.48	$0.05	$0.54	$0.08	**$5.71**
2033	$1.47	$2.60	$0.53	$1.26	**$5.85**	$0.41	$0.25	$1.96	$2.57	$0.05	$0.54	$0.08	**$5.85**
2034	$1.58	$2.58	$0.52	$1.25	**$5.93**	$0.41	$0.25	$1.94	$2.67	$0.05	$0.54	$0.08	**$5.93**
2035	$1.52	$2.57	$0.52	$1.24	**$5.86**	$0.41	$0.24	$1.93	$2.61	$0.05	$0.54	$0.08	**$5.86**
2036	$1.31	$2.52	$0.52	$1.24	**$5.58**	$0.40	$0.24	$1.86	$2.42	$0.05	$0.54	$0.07	**$5.58**
2037	$1.42	$2.46	$0.52	$1.23	**$5.63**	$0.41	$0.24	$1.79	$2.53	$0.05	$0.54	$0.08	**$5.63**
2038	$1.50	$2.52	$0.53	$1.29	**$5.83**	$0.43	$0.25	$1.84	$2.63	$0.05	$0.55	$0.08	**$5.83**
2039	$1.67	$2.69	$0.56	$1.42	**$6.34**	$0.50	$0.26	$2.03	$2.85	$0.05	$0.58	$0.08	**$6.34**
2040	$1.35	$2.54	$0.55	$1.39	**$5.83**	$0.47	$0.26	$1.84	$2.56	$0.05	$0.57	$0.08	**$5.83**
2041	$1.65	$2.80	$0.55	$1.38	**$6.39**	$0.48	$0.26	$2.17	$2.79	$0.05	$0.57	$0.08	**$6.39**
2042	$1.69	$2.90	$0.59	$1.52	**$6.70**	$0.53	$0.27	$2.26	$2.91	$0.05	$0.59	$0.08	**$6.70**
2043	$1.83	$3.24	$0.68	$1.92	**$7.67**	$0.71	$0.31	$2.58	$3.26	$0.05	$0.67	$0.09	**$7.67**
2044	$1.96	$3.29	$0.76	$2.29	**$8.30**	$0.78	$0.31	$2.88	$3.49	$0.05	$0.69	$0.10	**$8.30**
2045	$2.10	$3.51	$0.84	$2.61	**$9.06**	$0.81	$0.31	$3.47	$3.64	$0.05	$0.68	$0.10	**$9.06**
2046	$2.29	$3.68	$0.89	$2.83	**$9.69**	$0.95	$0.32	$3.68	$3.88	$0.05	$0.70	$0.10	**$9.69**
2047		$3.95	$0.66	$1.87	**$6.48**	$0.40	$0.28	$3.79	$1.27	$0.05	$0.61	$0.08	**$6.48**
2048		$3.91	$0.66	$1.86	**$6.44**	$0.40	$0.28	$3.74	$1.28	$0.05	$0.62	$0.08	**$6.44**
2049		$4.60	$0.83	$2.58	**$8.01**	$0.61	$0.34	$4.57	$1.58	$0.06	$0.76	$0.10	**$8.01**
2050													
Average	**$1.57**	**$2.60**	**$0.60**	**$1.36**	**$6.13**	**$0.46**	**$0.31**	**$1.94**	**$2.72**	**$0.05**	**$0.58**	**$0.08**	**$6.13**

21.2.2 Processing and G&A costs

Operating costs for the existing circuits

Operating costs for the oxide, mixed and sulphide circuits at Kansanshi are derived from actual costs incurred on site, and applied to the life of mine production schedule. These costs are presented in Table 21-7 and account for:

- crushing, milling, flotation, leaching and SX activities for the oxide circuit
- crushing, milling and flotation activities for the mixed and S2 sulphide circuits
- labour, fuel, consumables, reagents, power and maintenance costs for each circuit, and also separately in the HPL circuit and the gold plant.

Table 21-7 Kansanshi operating costs (life of mine averages) for the existing circuits

Circuit	Units	Fixed plus variable	Other direct	Total costs
Oxide (incl. SX)	$/t processed	$7.70	$2.49	$10.19
Mixed	$/t processed	$4.75	$2.46	$7.22
Sulphide S2	$/t processed	$4.84	$2.51	$7.35
Sulphide S3	$/t processed	$4.70	$2.38	$7.08
HPL	$/t conc.	$63.02		$63.02
Gold plant	$/oz Au	$107.87		$107.87

The HPL circuit currently handles approximately 130,000 t of flotation concentrate per year, oxidising the sulphides and producing a slurry containing ferric sulphate and copper sulphate in solution which is forwarded to the oxide leach circuit. This concentrate is material in excess of what can be smelted at the KCS facility and would otherwise be transported to third party smelters on the Copperbelt. The HPL processing circuit, therefore, is in place of KCS smelting and the processing costs are actually metal costs in terms of $/t of concentrate treated. The HPL product is excluded from the Mineral Reserve processing inventory.

Similarly, the gold plant costs in Table 21-7 are costs incurred for the upgrading of gravity concentrates to doré bars. Gravity concentrates are produced by all three circuits at Kansanshi and are treated separately in the gold plant for gold accounting purposes. The gold recovered in this way cannot be associated with the

Mineral Resource model (i.e. from which gold recovered into copper concentrate can be accounted for) and hence is excluded from the Mineral Reserve processing inventory.

S3 operating cost estimates

The S3 circuit is conceived as being one train of the Sentinel circuit. However, because S3 will be treating ore that is more similar to ores already being treated at Kansanshi, it is felt more appropriate to derive S3 operating costs from a combination of S2 and Sentinel costs.

In the 2020 Technical Report, the Sentinel direct process operating costs were estimated as \$5.06/t processed, and the derived S3 costs were \$4.79/t processed. S3 direct costs are now estimated to be \$4.71/t processed (at steady state). Table 21-8 shows the comparison of S2 and S3 operating cost estimates, at steady state five years hence. The S3 processing costs are higher than those for S2 processing due to the NaHS and depressant used in flotation.

Table 21-8 S2 and S3 process operating costs

Cost category	S2 (\$/t)	S3 (\$/t)
Employees	\$0.03	\$0.02
Contractors	\$0.01	\$0.10
Fuel	\$0.02	\$0.00
Consumables	\$0.16	\$0.07
Reagents	\$0.41	\$0.56
Energy	\$1.76	\$1.69
Maintenance	\$1.06	\$0.86
Crusher Liners	\$0.06	\$0.02
Milling Liners	\$0.36	\$0.56
Millballs	\$0.84	\$0.82
Other	\$0.00	\$0.01
Subtotal	**\$4.70**	**\$4.71**
Other direct	\$2.26	\$2.44
Total	**\$6.96**	**\$7.15**

These cost numbers represent typical years of operations and differ from the average life of mine costs given in Table 21-8, which also include high cost years at the beginning and end of operations at times when processed tonnages are low.

Energy costs

Energy costs were calculated from first principles using the Project equipment list and by estimating running times and power draw for each piece of equipment. An overall energy cost of 9.57c/kWh was applied yielding an estimated total energy unit cost for the S3 Project of \$1.69/t.

A cross check on this number using the Sentinel electricity cost, and the differences between one train of the Sentinel plant and S3 gave a similar cost.

Labour costs

Despite operating a single mill and flotation train at S3, vs two trains at Sentinel, labour numbers will not reduce to 50%, because the number of unit processes remains the same.

Staffing levels for S3 were estimated to be 75 process and 127 engineering personnel (compared with an estimate of 343 persons in the 2020 Technical Report), all of whom would be local hires (i.e. no expats). The average salary level would be \$14,000 pa, plus extras including bussing, pension contributions etc. This gives a total labour cost of \$0.11/t.

A cost of $0.02/t is given in Table 21-8; the remaining $0.09/t is included in the Other Direct Costs of $2.44/t shown in Table 21-7.

It should be noted that salaries are paid in Kwacha, whereas costs are quoted in US dollars. At the time of the 2020 Technical Report, the exchange rate was 18 Kwacha to the dollar; it is now 26 Kwacha to the dollar.

Reagents

Reagent costs for S3 have been calculated using actual reagent consumptions currently being experienced in the existing S2 sulphide circuit S2 at Kansanshi, together with current reagent prices.

Estimated reagent costs are shown in Table 21-9.

Table 21-9 Reagent costs for S3

Reagents	Consumption (g/t)	Cost of Reagent ($/t)	Treatment Cost ($/t)
Frother	10.8	3,723	0.08
Collector	13.5	6,285	0.08
NaHS for CPS	100.0	695	0.11
CMC depressant	50.0	4,040	0.20
Lime (pH modifier)	300.0	175	0.06
Flocculant (flotation tails)	20.0	2,226	0.04
			0.57

Minor reagents such as biocides and water treatment chemicals are included in Other Direct Costs, as they apply to all circuits, not just S3.

Steel ball costs

Steel ball consumption rates depend on the abrasion index of the ore and the power drawn by the mills (which is related to ore hardness).

Sentinel steel consumption (budget) is 0.22 kg/t of 140 mm balls for the same sized SAG mills. Using the relative abrasion indices for S3 and sentinel, this translates to 0.3kg/t of steel addition to the S3 SAG mill.

It is felt that the S3 ball mill steel consumption will be similar to that at S2, which is 0.22 kg/t of 65 mm balls.

Total consumption is thus 0.52 kg/t. S3 steel ball costs are thus estimated to be $0.83/t processed.

The consumption rate and cost is lower than estimated in the 2020 technical report due to optimisation of consumption rates at Sentinel (these benefits will flow onto S3), and a relaxation of the primary grind size at S3.

Maintenance consumables, maintenance, contractors and other costs

S3 will be 50% of the throughput of Sentinel, and will comprise a single milling and flotation circuit. However, it will contain the same unit processes and equipment sizes.

Therefore, one would expect to see half the maintenance costs in $ per annum terms, but the same costs in $/t terms. There are some differences in costs, however, because of some circuit differences:

- S3 will have a single primary crusher initially, with the second scheduled for the future, vs four at Sentinel
- S3 will have no secondary crushing

- S3 will have fewer rougher flotation cells, but the circuit will be more complex by inclusion of controlled sulphidisation (CPS)
- S3 will have two tailings thickeners, vs three at Sentinel
- S3 concentrate handling facilities are currently installed and operating at the smelter
- contractor costs may vary because of the skill sets available at Kansanshi

At the level of engineering detail available for the S3 circuit, the unit processing costs for Sentinel can be applied at Kansanshi, taking into account the above factors. The costs have also been benchmarked against S2 maintenance costs.

The 2020 Technical Report described maintenance, contractor and other costs as being \$1.57/t processed. These have been reduced to \$1.54/t processed, including an allowance of \$0.10 for contractors. By comparison S2 costs are estimated to be \$1.43/t, with only \$0.01/t for contractors.

Other Direct Costs

Other direct costs cover all items that are shared across the circuits. These items include process supervision, technical services, testwork, assay laboratory, supply and distribution of water, etc. The costs are the same for all circuits.

In 2024 Other Direct Costs are estimated to be \$5.34/t processed. These drop to \$2.24/t once S3 is fully operational and the costs are spread over 53 Mtpa instead of 28 Mtpa. Costs increase again to \$2.49/t processed in 2041 when only S3 is in operation.

The average of these costs over the life of mine is approximately \$2.38/t processed.

General and administration costs

General and administration costs relate to general management, security services, environmental management, commercial/finance functions, safety and support services, and reflects a combination of variable costs and (mostly) fixed costs.

These costs are currently being incurred and are not expected to increase when S3 is brought on line.

Over the remaining life of mine, G&A costs are expected to average \$1.55/t treated.

Summary of processing and G&A cost estimates

Table 21-10 lists the processing and G&A costs estimated for cashflow modelling, and profiled on an annual basis. The overall tonnes weighted average processing cost (excluding the gold and HPL plants) is \$7.48/t.

Table 21-10 Processing and G&A costs for cashflow modelling

Year	Oxide Circuit				Mixed Circuit			S2 Circuit			S3 Circuit			Gold plant	G&A costs
	Process $/t	SX $/t	Other direct $/t	Subtotal $/t	Process $/t	Other direct $/t	Subtotal $/t	Process $/t	Other direct $/t	Subtotal $/t	Process $/t	Other direct $/t	Subtotal $/t	$/oz	$/t
2024	$7.43	$0.86	$5.34	$13.63	$4.88	$5.34	$10.22	$4.62	$5.34	$9.97				$166.75	$1.80
2025	$6.44	$1.00	$2.90	$10.35	$4.61	$2.90	$7.51	$5.04	$2.90	$7.94	$3.44	$2.90	$6.34	$114.15	$1.62
2026	$6.69	$0.88	$2.27	$9.85	$5.03	$2.27	$7.30	$4.85	$2.27	$7.13	$5.07	$2.27	$7.34	$91.19	$1.41
2027	$6.71	$0.97	$2.27	$9.96	$3.82	$2.27	$6.09	$6.85	$2.27	$9.11	$4.69	$2.27	$6.95	$90.42	$1.42
2028	$6.71	$0.97	$2.29	$9.97	$4.89	$2.29	$7.18	$4.71	$2.29	$7.00	$4.74	$2.29	$7.03	$91.07	$1.43
2029	$6.71	$1.05	$2.29	$10.05	$4.89	$2.29	$7.18	$4.70	$2.29	$7.00	$4.73	$2.29	$7.03	$98.78	$1.42
2030	$6.71	$1.00	$2.29	$10.00	$4.89	$2.29	$7.18	$4.70	$2.29	$7.00	$4.73	$2.29	$7.03	$96.45	$1.42
2031	$6.71	$1.05	$2.29	$10.04	$4.89	$2.29	$7.18	$4.70	$2.29	$7.00	$4.73	$2.29	$7.03	$98.85	$1.42
2032	$6.70	$0.96	$2.24	$9.91	$4.89	$2.24	$7.13	$4.70	$2.24	$6.94	$4.71	$2.24	$6.95	$94.13	$1.36
2033	$6.71	$0.97	$2.24	$9.92	$4.89	$2.24	$7.13	$4.70	$2.24	$6.95	$4.71	$2.24	$6.95	$93.33	$1.37
2034	$6.71	$0.90	$2.24	$9.85	$4.89	$2.24	$7.13	$4.70	$2.24	$6.95	$4.71	$2.24	$6.95	$100.87	$1.37
2035	$6.71	$0.89	$2.24	$9.84	$4.89	$2.24	$7.13	$4.70	$2.24	$6.95	$4.71	$2.24	$6.95	$99.78	$1.37
2036	$6.70	$0.94	$2.24	$9.88	$4.89	$2.24	$7.13	$4.70	$2.24	$6.94	$4.71	$2.24	$6.95	$102.94	$1.36
2037	$6.71	$0.91	$2.24	$9.85	$4.89	$2.24	$7.13	$4.70	$2.24	$6.95	$4.71	$2.24	$6.95	$99.19	$1.37
2038	$6.71	$0.98	$2.25	$9.93	$4.69	$2.25	$6.94	$4.89	$2.25	$7.14	$4.71	$2.25	$6.96	$92.07	$1.37
2039	$6.71	$1.04	$2.25	$10.00	$4.69	$2.25	$6.94	$4.89	$2.25	$7.14	$4.71	$2.25	$6.96	$101.14	$1.37
2040											$4.71	$2.83	$7.54	$117.76	$2.10
2041											$4.71	$2.49	$7.20	$126.70	$1.99
2042											$4.71	$2.49	$7.20	$119.10	$1.87
2043											$4.71	$2.49	$7.20	$125.47	$1.87
2044											$4.71	$2.48	$7.20	$138.46	$1.87
2045											$4.71	$2.49	$7.20	$139.72	$1.87
2046											$4.71	$2.49	$7.20	$129.17	$1.87
2047											$4.71	$2.49	$7.20	$170.76	$1.87
2048											$4.71	$2.48	$7.20	$170.76	$1.87
2049											$4.83	$2.33	$7.15	$117.22	$2.73
2050															
Average	**$6.73**	**$0.96**	**$2.49**	**$10.19**	**$4.75**	**$2.46**	**$7.22**	**$4.84**	**$2.51**	**$7.35**	**$4.70**	**$2.38**	**$7.08**	**$107.87**	**$1.55**

21.2.3 Metal costs

The metal costs required for cashflow modelling were derived from actual incurred costs, as itemised in Table 21-11.

The cathode EW costs account for fixed and variable charges, for labour, fuel, consumables, power and maintenance. Cathode freight charges include land transport and shipping.

The KCS smelting costs include the treatment charge for concentrate received into the smelter, plus the anode refining charge. Concentrate is pumped to the smelter from the process plant; hence a land transport charge does not apply. Anode freight charges include land transport and shipping.

Table 21-11 Metal costs for cashflow modelling

Transport, smelting and refining charges	Units	$/unit
Cathode EW costs		
Electrowinning	$/t cathode	395.86
	$/lb Cu	0.18
Cathode freight	$/t cathode	263.40
	$/lb Cu	0.12
Cathode insurance	$/t cathode	1.86
	$/lb Cu	0.00
Total cathode cost	**$/t cathode**	**661.12**
	$/lb Cu	**0.30**
Smelting/refining costs		
Concentrate grade	%	24.3
Concentrate road haualge to KCS	$/t con.	0.00
	$/lb Cu	0.00
KCS smelting cost	$/t con.	89.57
	$/lb Cu	0.17
Anode transport cost	$/t metal	218.58
	$/lb Cu	0.10
Insurance	$/t metal	1.84
	$/lb Cu	0.00
Refining charge	$/t metal	123.48
	$/lb Cu	0.06
Total smelting/refining cost	**$/lb Cu**	**0.32**
Gold refining charge	**$/lb oz**	**5.00**

ITEM 22 ECONOMIC ANALYSIS

22.1 Principal assumptions

In accordance with NI 43-101 and Companion Policy 43-101CP, the economic analysis set out below does not include Inferred Mineral Resource as a revenue source.

The economic analysis in the form of a 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 for completion of the S3 processing plant and for expansion of the KCS are included in the model for completeness, as are sustaining costs and longer-term rehabilitation provisions. The cashflow model commencement year is 2024, and the cashflows are shown pre-tax and post-tax.

22.1.1 Metal pricing

The annual revenues in a base case cashflow model referenced the same metal prices as adopted for the pit optimisation (Item 15.4.1), namely $3.50/lb Cu and $1,805/oz Au.

An additional sensitivity analysis on the cashflow model referenced April 2024 average consensus pricing information, from a number of banks and financial service institutions, as listed in Table 22-1. In this particular analysis, a long-term price of $4.02/lb copper was adopted for sensitivity modelling.

22.1.2 Royalties and other levies

Further to the information provided in Item 4.4, the Mineral Reserve cashflow model adopts a sliding scale royalty on copper production, averaging about 6% over the life of the Project, and a 6% royalty on gold production. Apart from a 3.1% royalty to ZCCM-IH, there are no other applicable levies or related charges carried in the cashflow model.

22.1.3 Taxation

The cashflow model is reported in pre-tax terms, and also in post-tax terms reflecting a 30% corporate taxation rate. The modelled taxes and royalty payments are net of value added tax (VAT) refunds owed to the Company in accordance with a VAT agreement with the Zambian government.

22.2 Cashflow model inputs

22.2.1 Production schedule

The production schedule forming the basis of the Mineral Reserve cashflow model is the same as that listed in the various tables of Item 16. The process recoveries for copper vary for each processing circuit according to equations developed from metallurgical performance analysis and projections. The overall average copper recoveries in the model are as follows:

- sulphide ore flotation = 87.8%
- mixed ore flotation = 67.7%
- oxide flotation = 32.7%
- oxide leach = 40.4%

Fixed gold recoveries are modelled as 37.5% in the sulphide ore flotation, 32.0% in the mixed ore flotation and 24.0 % in the oxide ore flotation.

Table 22-1 Copper and gold pricing information referenced for cashflow sensitivity modelling

Commodity	Units	Date	2024	2025	2026	2027	2028	LT
Copper	$/lb	**17-Apr-24**	$4.01	$4.29	$4.33	$4.38	$4.33	$4.02
Gold	$/oz	**17-Apr-24**	$2,068	$2,072	$1,961	$1,879	$1,903	$1,805

In terms of KCS smelting, a recovery of 97.8% is adopted in the model.

For the Mineral Reserves cashflow model, it is assumed that no Kansanshi copper concentrate is processed through the HPL circuit.

22.2.2 Revenue inputs

In addition to the metal pricing information in Table 22-1 and, other revenue related inputs to the model are listed below.

The payable metal factors are:

- cathode copper = 100%
- anode copper = 99.7%
- gold in concentrate (anode) = 90.7%

22.2.3 Capital and sustaining cost inputs

According to the itemisation provided in Table 21-1, a total Project capital cost of $1,68.7 M was included in the cashflow model commencing from 2024, along with an additional $2,319.6 M of sustaining cost provisions for the period 2024 to 2049 (Table 21-3).

An amount of $128.2 M is included for Project closure costs, spent over the final years of Project life and accounting for the early removal of redundant processing infrastructure.

22.2.4 Operating and metal cost inputs

The estimation methodology for annual unit mining costs is summarised in Item 21.2.1, with the varying rates as listed in Table 21-5 thereby adopted for Mineral Reserve cashflow modelling. The overall average unit mining costs are:

- ore mined to the plant direct and to stockpiles = $6.13/bcm
- ore reclaimed from stockpiles = $5.30/bcm
- waste mined to dump = $6.08/bcm

From the information in Table 21-10, profiled annual processing and G&A costs were included in the cashflow model, yielding overall average unit costs of:

- oxide ore processing = $10.19/t
- mixed ore processing = $7.22/t
- S2 sulphide ore processing = $7.35/t
- S3 sulphide ore processing = $7.08/t
- G&A = $1.55/t

The modelled metal costs (including TCRC's and royalties) equate to the same costs as summarised in Item 21.2.3:

- cathode EW = $0.30/lb

- copper concentrate, freight and refining = $0.32/lb
- gold refining = $5.00/oz
- copper royalty = $0.21/lb Cu
- gold royalty = $108.3/oz Au
- ZCCM royalty = $0.11/lb Cu (including contribution from gold)

22.3 Cashflow model outcomes

The base case cashflow model to support the Mineral Reserve estimate is listed in Table 22-2.

The pre-tax undiscounted cashflow for the Mineral Reserve production schedule is $8.2 B as from 2024, with an NPV reflecting a 10% discount rate equal to $2.9 B. On a post-tax basis, the undiscounted cashflow is $6.3 B as from 2024, with an NPV equal to $2.2 B. The corresponding internal rate of return (IRR) from 2024 is 42%.

When the revenue and costs associated with the additional gold are included (i.e., outside of the Mineral Reserve inventory), the pre-tax undiscounted cashflow and NPV figures rise to $9.6 B and $3.5 B, respectively. Post-tax, these figures are $7.3 B and $2.7 B, respectively. The corresponding IRR from 2024 is 50%.

If an April 2024 consensus, long term copper price of $4.02/lb was adopted, the pre-tax undiscounted cashflow and NPV under these circumstances, would be $13.3 B and $5.2 B, respectively. Post-tax, the undiscounted cashflow and NPV would be $9.9 B and $4.0 B, respectively, and the IRR (from 2024) would be 89%.

At the higher copper price and with the inclusion of additional gold outside of the Mineral Reserve inventory, the pre-tax cashflow and NPV figures become $14.8 B and $5.8 B, respectively. Post-tax, the respective figures are $10.9 B and $4.4 B, with an IRR (from 2024) of 104%.

These figures compare with the pre-tax and post-tax modelling outcomes listed in Table 22-3.

At the Mineral Reserve copper price of $3.50/lb Cu, under post-tax circumstances and starting from 2024, the payback year is 2028. At the higher copper price, the payback year would be 2026.

22.4 Project value and sensitivity analysis

A sensitivity analysis was completed as part of the pit optimisation work described in Item 15.4.3. A subsequent analysis was completed on the pre-tax cashflow, the inputs and the analysis results from which are listed In Table 22-4. From this table, the following inferences can be drawn:

- the value of the additional gold (outside of the Mineral Reserve inventory) could contribute significantly to Project value, more so than less likely increases or reductions to process recovery and costs, respectively
- in the case of value detractions, capex increases by up to 20% could lead to a 10% reduction in NPV
- the magnitude of the NPV reduction would be similar for metal cost increases by up to 20%
- mining and processing cost increases by up to 20% have a greater impact on NPV
- reduced recovery projections would have the greatest impact on NPV, obviously, since this would be analogous to a commensurate reduction in feed grade

Table 22-2 Mineral Reserve cashflow model

PHYSICALS	UNITS	FROM 2024	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050
MINING																														
Total ore	**Mbcm**	**351.6**		10.9	19.6	18.5	17.4	20.4	21.0	23.0	19.8	19.2	21.2	21.9	21.5	19.2	16.5	17.4	12.3	11.6	6.1	8.5	10.3	7.7	4.1	3.8	0.0	0.0	0.0	0.0
Total waste	**Mbcm**	**1,069.3**		38.4	48.8	50.1	58.5	57.4	58.3	56.2	60.3	61.1	58.8	58.4	58.6	61.2	62.4	57.4	46.7	58.8	41.4	35.3	22.6	13.4	3.2	2.1	0.0	0.0	0.0	0.0
Total mined	**Mbcm**	**1,420.8**		**49.3**	**68.4**	**68.6**	**75.9**	**77.8**	**79.3**	**79.2**	**80.1**	**80.3**	**79.9**	**80.3**	**80.1**	**80.4**	**78.9**	**74.8**	**59.0**	**70.3**	**47.4**	**43.8**	**32.8**	**21.1**	**7.3**	**5.9**	**0.0**	**0.0**	**0.0**	**0.0**
Additional material movement																														
stockpile relocation	**Mbcm**	**12.7**		4.9	7.5	0.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Total ore	**Mt**	**935.2**		27.7	46.1	45.4	44.2	49.9	53.4	61.6	53.9	52.3	57.9	60.3	59.0	52.1	45.1	48.0	33.5	32.0	16.8	23.7	28.7	21.6	11.4	10.6	0.0	0.0	0.0	0.0
Total waste	**Mt**	**2,718.3**		75.8	107.9	116.3	134.4	139.0	153.4	147.6	157.8	154.1	151.4	161.8	158.7	151.6	158.1	149.4	128.1	144.9	113.3	98.8	63.1	37.7	9.2	5.9	0.0	0.0	0.0	0.0
Total mined	**Mt**	**3,653.6**		**103.5**	**153.9**	**161.7**	**178.6**	**188.9**	**206.7**	**209.2**	**211.8**	**206.4**	**209.4**	**222.1**	**217.8**	**203.7**	**203.2**	**197.3**	**161.6**	**176.9**	**130.1**	**122.5**	**91.8**	**59.3**	**20.6**	**16.5**	**0.0**	**0.0**	**0.0**	**0.0**
ore density	t/bcm	**2.66**		2.55	2.36	2.46	2.54	2.44	2.54	2.68	2.72	2.73	2.74	2.75	2.74	2.72	2.74	2.75	2.73	2.76	2.77	2.79	2.79	2.81	2.81	2.79	0.00	0.00	0.00	0.00
waste density	t/bcm	**2.54**		1.97	2.21	2.32	2.30	2.42	2.63	2.63	2.62	2.52	2.58	2.77	2.71	2.48	2.53	2.60	2.75	2.46	2.74	2.80	2.80	2.81	2.83	2.80	0.00	0.00	0.00	0.00
total density	t/bcm	**2.57**		2.10	2.25	2.36	2.35	2.43	2.61	2.64	2.64	2.57	2.62	2.77	2.72	2.54	2.58	2.64	2.74	2.51	2.74	2.80	2.79	2.81	2.82	2.80	0.00	0.00	0.00	0.00
Strip ratio		**2.9**		2.7	2.3	2.6	3.0	2.8	2.9	2.4	2.9	2.9	2.6	2.7	2.7	2.9	3.5	3.1	3.8	4.5	6.8	4.2	2.2	1.7	0.8	0.6	0.0	0.0	0.0	0.0
STOCKPILE MOVEMENTS																														
Total on	**Mt**	**169.6**		14.8	24.2	18.4	10.4	16.0	16.9	13.3	6.1	3.0	5.8	9.8	10.0	4.9	5.0	1.7	0.0	4.5	1.3	0.5	1.7	1.4	0.0	0.0	0.0	0.0	0.0	0.0
	%	**0.37**		0.46	0.46	0.33	0.33	0.35	0.35	0.34	0.37	0.33	0.32	0.30	0.31	0.39	0.35	0.30	0.00	0.31	0.30	0.28	0.32	0.30	0.00	0.00	0.00	0.00	0.00	0.00
Total off	**Mt**	**339.1**		15.7	20.6	26.5	19.6	19.7	17.1	5.3	5.7	6.9	3.9	5.4	7.0	8.9	15.8	9.5	22.2	0.0	11.9	4.2	0.4	7.3	15.9	16.8	27.4	27.5	18.1	0.0
	%	**0.38**		0.60	0.52	0.40	0.41	0.46	0.35	0.39	0.38	0.37	0.40	0.39	0.36	0.38	0.36	0.47	0.32	0.00	0.29	0.29	0.29	0.29	0.29	0.30	0.33	0.33	0.33	0.00
Total balance	**Mt**		**169.5**	**168.5**	**172.1**	**164.0**	**154.8**	**151.0**	**150.8**	**158.8**	**159.1**	**155.3**	**157.3**	**161.6**	**164.7**	**160.7**	**149.8**	**142.1**	**119.9**	**124.4**	**113.8**	**110.2**	**111.5**	**105.6**	**89.7**	**72.9**	**45.5**	**18.0**	**0.0**	**0.0**
	%		**0.40**	**0.38**	**0.38**	**0.37**	**0.36**	**0.35**	**0.35**	**0.35**	**0.35**	**0.35**	**0.34**	**0.34**	**0.34**	**0.34**	**0.33**	**0.32**	**0.33**	**0.32**	**0.33**	**0.33**	**0.33**	**0.33**	**0.34**	**0.35**	**0.36**	**0.39**	**0.00**	**0.00**
TOTAL FEED TO PLANT																														
Oxide flotation/leach	Mt	**116.8**		7.3	7.3	7.2	7.2	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	7.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	%TCu	**0.60**		0.63	0.65	0.59	0.56	0.60	0.75	0.69	0.75	0.61	0.61	0.45	0.45	0.55	0.46	0.59	0.70	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	g/tAu	**0.12**		0.14	0.15	0.14	0.16	0.10	0.12	0.11	0.11	0.12	0.11	0.08	0.08	0.09	0.10	0.11	0.11	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	kt	**703.3**		46.1	47.3	42.5	40.9	43.5	55.1	50.1	55.1	44.9	44.7	32.7	32.8	40.0	33.8	42.9	51.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	k(t)oz	**434.7**		32.8	36.0	33.2	36.3	23.6	28.9	26.2	26.8	28.5	25.7	19.3	19.9	21.6	22.4	26.9	26.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Mixed flotation	Mt	**141.4**		7.8	7.8	7.8	12.4	7.8	7.8	7.8	7.8	7.8	7.8	7.8	7.8	7.8	7.8	13.9	13.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	%TCu	**0.65**		0.81	1.06	0.75	0.59	0.55	0.62	0.90	0.76	0.70	0.75	0.63	0.65	0.63	0.60	0.48	0.37	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	g/tAu	**0.12**		0.22	0.26	0.13	0.11	0.10	0.10	0.15	0.13	0.12	0.14	0.10	0.11	0.10	0.10	0.07	0.08	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	kt	**924.1**		63.4	82.5	58.2	73.5	43.2	47.9	69.9	59.2	54.5	58.2	49.3	50.2	48.8	46.8	66.9	51.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	k(t)oz	**554.2**		54.2	65.9	33.3	43.9	24.1	25.9	38.3	32.2	30.8	34.8	25.6	27.6	24.8	25.2	33.1	34.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Sulphide S2 & S3 flotation	Mt	**846.6**		13.6	27.4	38.5	33.7	38.6	38.5	38.5	38.5	41.0	40.9	40.9	40.9	41.0	40.9	34.5	34.5	27.5	27.4	27.4	27.4	27.4	27.4	27.4	27.4	27.5	18.1	0.0
	%TCu	**0.49**		0.64	0.49	0.41	0.54	0.55	0.50	0.54	0.56	0.56	0.56	0.55	0.55	0.52	0.51	0.56	0.47	0.55	0.44	0.54	0.47	0.41	0.31	0.31	0.33	0.33	0.33	0.00
	g/tAu	**0.10**		0.15	0.12	0.09	0.10	0.11	0.10	0.10	0.10	0.11	0.11	0.10	0.10	0.10	0.10	0.13	0.10	0.12	0.10	0.11	0.10	0.09	0.09	0.10	0.07	0.07	0.07	0.00
	kt	**4,144.2**		86.3	133.8	159.5	182.2	210.8	190.8	207.6	214.9	228.5	228.4	224.4	226.2	213.4	207.0	194.4	161.5	150.5	119.7	149.1	130.0	111.8	85.7	84.5	91.3	91.6	60.2	0.0
	k(t)oz	**2,779.1**		64.0	109.4	116.7	111.3	130.8	125.4	128.0	123.2	147.9	150.5	134.9	136.5	127.8	137.7	139.1	111.1	102.1	90.5	99.9	91.9	79.2	77.9	87.8	58.5	58.6	38.6	0.0
Total plant feed	**Mt**	**1,104.8**		**28.7**	**42.5**	**53.5**	**53.4**	**53.7**	**53.6**	**53.6**	**53.6**	**56.1**	**56.0**	**56.0**	**56.0**	**56.1**	**56.0**	**55.7**	**55.7**	**27.5**	**27.4**	**27.4**	**27.4**	**27.4**	**27.4**	**27.4**	**27.4**	**27.5**	**18.1**	**0.0**
	%TCu	**0.52**		**0.68**	**0.62**	**0.49**	**0.56**	**0.55**	**0.55**	**0.61**	**0.61**	**0.58**	**0.59**	**0.55**	**0.55**	**0.54**	**0.51**	**0.55**	**0.47**	**0.55**	**0.44**	**0.54**	**0.47**	**0.41**	**0.31**	**0.31**	**0.33**	**0.33**	**0.33**	**0.00**
	g/tAu	**0.11**		**0.16**	**0.15**	**0.11**	**0.11**	**0.10**	**0.10**	**0.11**	**0.11**	**0.11**	**0.12**	**0.10**	**0.10**	**0.10**	**0.10**	**0.11**	**0.10**	**0.12**	**0.10**	**0.11**	**0.10**	**0.09**	**0.09**	**0.10**	**0.07**	**0.07**	**0.07**	**0.00**
	kt	**5,771.6**		**195.9**	**263.6**	**260.1**	**296.6**	**297.5**	**293.8**	**327.6**	**329.2**	**327.9**	**331.3**	**306.4**	**309.1**	**302.3**	**287.6**	**304.2**	**264.2**	**150.5**	**119.7**	**149.1**	**130.0**	**111.8**	**85.7**	**84.5**	**91.3**	**91.6**	**60.2**	**0.0**
	k(t)oz	**3,768.0**		**151.0**	**211.3**	**183.2**	**191.5**	**178.5**	**180.2**	**192.5**	**182.2**	**207.1**	**211.0**	**179.8**	**184.1**	**174.1**	**185.4**	**199.0**	**172.2**	**102.1**	**90.5**	**99.9**	**91.9**	**79.2**	**77.9**	**87.8**	**58.5**	**58.6**	**38.6**	**0.0**
AVERAGE RECOVERIES																														
Oxide leach	Cu	**40.4%**		43.8%	40.2%	43.1%	41.4%	40.0%	43.2%	39.9%	42.4%	37.9%	38.8%	36.0%	33.0%	37.7%	35.7%	42.3%	45.2%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Oxide flotation	Cu	**32.7%**		31.3%	34.0%	31.4%	31.9%	33.0%	31.4%	33.2%	31.9%	34.2%	33.5%	34.6%	36.6%	33.9%	34.8%	31.1%	29.8%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
	Au	**24.0%**		24.0%	24.0%	24.0%	24.0%	24.0%	24.0%	24.0%	24.0%	24.0%	24.0%	24.0%	24.0%	24.0%	24.0%	24.0%	24.0%	24.0%	24.0%	24.0%	24.0%	24.0%	24.0%	24.0%	24.0%	24.0%	24.0%	0.0%
Mixed flotation	%	**67.7%**		71.1%	73.3%	68.7%	66.8%	64.0%	64.5%	69.9%	68.3%	68.8%	68.9%	67.9%	67.9%	66.8%	66.5%	64.0%	60.3%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
	%	**32.0%**		32.0%	32.0%	32.0%	32.0%	32.0%	32.0%	32.0%	32.0%	32.0%	32.0%	32.0%	32.0%	32.0%	32.0%	32.0%	32.0%	32.0%	32.0%	32.0%	32.0%	32.0%	32.0%	32.0%	32.0%	32.0%	32.0%	0.0%
Sulphide S2 & S3 flotation	%	**87.8%**		90.2%	89.4%	89.1%	89.4%	89.9%	89.2%	89.3%	89.6%	89.6%	89.5%	90.4%	90.4%	90.1%	89.7%	90.1%	87.9%	90.1%	85.5%	89.5%	90.2%	85.8%	79.8%	77.9%	68.4%	68.4%	68.4%	0.0%
	%	**37.5%**		37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	37.5%	0.0%
Overall average	**%**	**82.8%**		**80.5%**	**81.6%**	**82.2%**	**81.6%**	**83.7%**	**82.4%**	**82.7%**	**83.2%**	**83.7%**	**83.5%**	**84.6%**	**84.6%**	**83.9%**	**83.6%**	**82.0%**	**80.0%**	**90.1%**	**85.5%**	**89.5%**	**90.2%**	**85.8%**	**79.8%**	**77.9%**	**68.4%**	**68.4%**	**68.4%**	**0.0%**
	%	**35.1%**		**32.6%**	**33.5%**	**34.1%**	**33.7%**	**35.0%**	**34.5%**	**34.6%**	**34.5%**	**34.8%**	**34.9%**	**35.3%**	**35.2%**	**35.0%**	**35.1%**	**34.8%**	**34.3%**	**37.5%**	**37.5%**	**37.5%**	**37.5%**	**37.5%**	**37.5%**	**37.5%**	**37.5%**	**37.5%**	**37.5%**	**0.0%**
METAL RECOVERED																														
Oxide leach	kt	**284.2**		20.2	19.0	18.3	16.9	17.4	23.8	20.0	23.3	17.0	17.3	11.8	10.8	15.1	12.0	18.2	23.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Oxide flotation	kt	**230.3**		14.4	16.0	13.3	13.0	14.4	17.3	16.7	17.6	15.3	15.0	11.3	12.0	13.6	11.8	13.3	15.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	k(t)oz	**104.3**		7.9	8.6	8.0	8.7	5.7	6.9	6.3	6.4	6.8	6.2	4.6	4.8	5.2	5.4	6.5	6.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Mixed flotation	kt	**625.2**		45.1	60.5	40.0	49.2	27.6	30.9	48.8	40.4	37.5	40.1	33.4	34.1	32.6	31.1	42.8	31.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	k(t)oz	**177.3**		17.3	21.1	10.7	14.1	7.7	8.3	12.3	10.3	9.8	11.2	8.2	8.8	7.9	8.1	10.6	11.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Sulphide S2 & S3 flotation	kt	**3,636.6**		77.9	119.6	142.1	162.8	189.6	170.2	185.4	192.5	204.8	204.4	202.8	204.5	192.4	185.6	175.2	142.0	135.6	102.3	133.5	117.2	96.0	68.4	65.8	62.5	62.6	41.2	0.0
	k(t)oz	**1,042.2**		24.0	41.0	43.8	41.7	49.1	47.0	48.0	46.2	55.4	56.4	50.6	51.2	47.9	51.7	52.1	41.7	38.3	33.9	37.5	34.5	29.7	29.2	32.9	21.9	22.0	14.5	0.0
Total metal recovered	**kt**	**4,776.3**		**157.6**	**215.1**	**213.7**	**241.9**	**248.9**	**242.2**	**270.8**	**273.8**	**274.6**	**276.8**	**259.3**	**261.4**	**253.6**	**240.5**	**249.6**	**211.3**	**135.6**	**102.3**	**133.5**	**117.2**	**96.0**	**68.4**	**65.8**	**62.5**	**62.6**	**41.2**	**0.0**
	k(t)oz	**1,323.8**		**49.2**	**70.7**	**62.4**	**64.5**	**62.4**	**62.2**	**66.5**	**62.9**	**72.1**	**73.7**	**63.4**	**64.8**	**61.0**	**65.1**	**69.2**	**59.1**	**38.3**	**33.9**	**37.5**	**34.5**	**29.7**	**29.2**	**32.9**	**21.9**	**22.0**	**14.5**	**0.0**

PAYABILITY AND REVENUE SUMMARY	UNITS	FROM 2024	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050
PAYABLE COPPER																														
Copper in cathode																														
Cathode produced from oxide	kt	**284.2**		20.2	19.0	18.3	16.9	17.4	23.8	20.0	23.3	17.0	17.3	11.8	10.8	15.1	12.0	18.2	23.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Total cathode produced	**kt**	**284.2**		**20.2**	**19.0**	**18.3**	**16.9**	**17.4**	**23.8**	**20.0**	**23.3**	**17.0**	**17.3**	**11.8**	**10.8**	**15.1**	**12.0**	**18.2**	**23.1**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**
Copper in concentrate/anode																														
from oxide feed	kt	**230.3**		14.4	16.0	13.3	13.0	14.4	17.3	16.7	17.6	15.3	15.0	11.3	12.0	13.6	11.8	13.3	15.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
from mixed feed	kt	**625.2**		45.1	60.5	40.0	49.2	27.6	30.9	48.8	40.4	37.5	40.1	33.4	34.1	32.6	31.1	42.8	31.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
from sulphide feed	kt	**3,636.6**		77.9	119.6	142.1	162.8	189.6	170.2	185.4	192.5	204.8	204.4	202.8	204.5	192.4	185.6	175.2	142.0	135.6	102.3	133.5	117.2	96.0	68.4	65.8	62.5	62.6	41.2	0.0
total excl. cathode production	kt	**4,492.1**		**137.4**	**196.1**	**195.4**	**225.0**	**231.6**	**218.4**	**250.9**	**250.5**	**257.6**	**259.4**	**247.5**	**250.6**	**238.5**	**228.5**	**231.4**	**188.2**	**135.6**	**102.3**	**133.5**	**117.2**	**96.0**	**68.4**	**65.8**	**62.5**	**62.6**	**41.2**	**0.0**
concentrate grade	**%**	**23.4%**		**22.5%**	**22.8%**	**22.7%**	**22.7%**	**23.3%**	**23.0%**	**22.7%**	**22.9%**	**23.2%**	**23.1%**	**23.4%**	**23.3%**	**23.3%**	**23.3%**	**22.9%**	**22.8%**	**25.0%**	**25.0%**	**25.0%**	**25.0%**	**25.0%**	**25.0%**	**25.0%**	**25.0%**	**25.0%**	**25.0%**	**25.0%**
concentrate to process:																														
to KCS	kt	**19,211.3**		612.0	860.5	859.8	992.6	992.4	948.0	1,104.7	1,091.5	1,111.9	1,122.8	1,059.1	1,073.5	1,025.4	980.2	1,012.4	824.4	542.3	409.1	533.8	468.9	384.0	273.5	263.3	249.8	250.5	164.7	0.0
Total concentrate produced	**kt**	19,211.3		**612.0**	**860.5**	**859.8**	**992.6**	**992.4**	**948.0**	**1,104.7**	**1,091.5**	**1,111.9**	**1,122.8**	**1,059.1**	**1,073.5**	**1,025.4**	**980.2**	**1,012.4**	**824.4**	**542.3**	**409.1**	**533.8**	**468.9**	**384.0**	**273.5**	**263.3**	**249.8**	**250.5**	**164.7**	**0.0**
KCS product																														
KCS smelter recovery	kt	**18,796.3**		598.7	841.9	841.2	971.2	971.0	927.5	1080.8	1067.9	1087.9	1098.6	1036.3	1050.3	1003.3	959.0	990.6	806.6	530.6	400.2	522.3	458.8	375.7	267.6	257.6	244.4	245.1	161.2	0.0
Process Loss	kt	**415.0**		13.2	18.6	18.6	21.4	21.4	20.5	23.9	23.6	24.0	24.3	22.9	23.2	22.1	21.2	21.9	17.8	11.7	8.8	11.5	10.1	8.3	5.9	5.7	5.4	5.4	3.6	0.0
Total anode produced	**kt**	4,375.3		**133.9**	**191.0**	**190.3**	**219.2**	**225.5**	**212.7**	**244.3**	**244.0**	**250.9**	**252.7**	**241.1**	**244.1**	**232.3**	**222.5**	**225.4**	**183.4**	**132.1**	**99.6**	**130.0**	**114.2**	**93.5**	**66.6**	**64.1**	**60.8**	**61.0**	**40.1**	**0.0**
Total copper metal produced																														
copper in cathode	kt	**284.2**		20.2	19.0	18.3	16.9	17.4	23.8	20.0	23.3	17.0	17.3	11.8	10.8	15.1	12.0	18.2	23.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
copper in anode	kt	**4,375.3**		133.9	191.0	190.3	219.2	225.5	212.7	244.3	244.0	250.9	252.7	241.1	244.1	232.3	222.5	225.4	183.4	132.1	99.6	130.0	114.2	93.5	66.6	64.1	60.8	61.0	40.1	0.0
Total Cu metal produced	**kt**	4,659.5		**154.1**	**210.0**	**208.6**	**236.1**	**242.9**	**236.5**	**264.3**	**267.3**	**267.9**	**270.0**	**252.8**	**254.9**	**247.4**	**234.6**	**243.5**	**206.4**	**132.1**	**99.6**	**130.0**	**114.2**	**93.5**	**66.6**	**64.1**	**60.8**	**61.0**	**40.1**	**0.0**
Total payable metal																														
copper in cathode	kt	**284.2**		20.2	19.0	18.3	16.9	17.4	23.8	20.0	23.3	17.0	17.3	11.8	10.8	15.1	12.0	18.2	23.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
copper in anode	kt	**4,362.1**		133.5	190.5	189.8	218.5	224.9	212.1	243.6	243.2	250.1	251.9	240.3	243.3	231.6	221.9	224.7	182.8	131.7	99.3	129.6	113.8	93.2	66.4	63.9	60.7	60.8	40.0	0.0
Total Cu metal sold	**kt**	4,646.4		**153.7**	**209.5**	**208.1**	**235.4**	**242.2**	**235.9**	**263.6**	**266.6**	**267.1**	**269.3**	**252.1**	**254.1**	**246.7**	**233.9**	**242.9**	**205.9**	**131.7**	**99.3**	**129.6**	**113.8**	**93.2**	**66.4**	**63.9**	**60.7**	**60.8**	**40.0**	**0.0**
PAYABLE GOLD																														
Gold in concentrate																														
from oxide circuit	k(t)oz	**104.3**		7.9	8.6	8.0	8.7	5.7	6.9	6.3	6.4	6.8	6.2	4.6	4.8	5.2	5.4	6.5	6.4	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
from mixed circuit	k(t)oz	**177.3**		17.3	21.1	10.7	14.1	7.7	8.3	12.3	10.3	9.8	11.2	8.2	8.8	7.9	8.1	10.6	11.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
from sulphide circuit	k(t)oz	**1,042.2**		24.0	41.0	43.8	41.7	49.1	47.0	48.0	46.2	55.4	56.4	50.6	51.2	47.9	51.7	52.1	41.7	38.3	33.9	37.5	34.5	29.7	29.2	32.9	21.9	22.0	14.5	0.0
Total gold in Concentrate	**k(t)oz**	**1,323.8**		49.2	70.7	62.4	64.5	62.4	62.2	66.5	62.9	72.1	73.7	63.4	64.8	61.0	65.1	69.2	59.1	38.3	33.9	37.5	34.5	29.7	29.2	32.9	21.9	22.0	14.5	0.0
Total gold produced	**k(t)oz**	1,323.8		**49.2**	**70.7**	**62.4**	**64.5**	**62.4**	**62.2**	**66.5**	**62.9**	**72.1**	**73.7**	**63.4**	**64.8**	**61.0**	**65.1**	**69.2**	**59.1**	**38.3**	**33.9**	**37.5**	**34.5**	**29.7**	**29.2**	**32.9**	**21.9**	**22.0**	**14.5**	**0.0**
Total payable gold																														
gravity gold	k(t)oz	**0.0**		**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**
gold in concentrate	k(t)oz	**1,295.2**		**48.2**	**69.2**	**61.1**	**63.1**	**61.1**	**60.9**	**65.1**	**61.6**	**70.6**	**72.1**	**62.0**	**63.4**	**59.7**	**63.7**	**67.7**	**57.8**	**37.4**	**33.2**	**36.7**	**33.7**	**29.0**	**28.6**	**32.2**	**21.5**	**21.5**	**14.1**	**0.0**
gold in anode	k(t)oz	**1,173.3**		**44.1**	**63.7**	**56.2**	**58.0**	**56.2**	**55.2**	**59.3**	**55.6**	**64.1**	**65.8**	**56.0**	**57.4**	**53.9**	**57.5**	**61.3**	**51.6**	**33.9**	**29.9**	**33.2**	**30.3**	**25.8**	**25.6**	**29.2**	**18.5**	**18.6**	**12.3**	**0.0**
Total Au metal sold	**k(t)oz**	1,173.3		**44.1**	**63.7**	**56.2**	**58.0**	**56.2**	**55.2**	**59.3**	**55.6**	**64.1**	**65.8**	**56.0**	**57.4**	**53.9**	**57.5**	**61.3**	**51.6**	**33.9**	**29.9**	**33.2**	**30.3**	**25.8**	**25.6**	**29.2**	**18.5**	**18.6**	**12.3**	**0.0**
GROSS REVENUE																														
Metal prices	**variable**																													
Copper	Cu $/lb			$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50
Copper	Cu $/t			$7,716	$7,716	$7,716	$7,716	$7,716	$7,716	$7,716	$7,716	$7,716	$7,716	$7,716	$7,716	$7,716	$7,716	$7,716	$7,716	$7,716	$7,716	$7,716	$7,716	$7,716	$7,716	$7,716	$7,716	$7,716	$7,716	$7,716
Gold	Au $/oz			$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805
Copper revenue	**$M**	**$35,852.1**		**$1,185.7**	**$1,616.3**	**$1,605.5**	**$1,816.6**	**$1,869.2**	**$1,820.3**	**$2,033.8**	**$2,056.8**	**$2,061.3**	**$2,077.7**	**$1,945.3**	**$1,961.0**	**$1,903.8**	**$1,804.8**	**$1,874.0**	**$1,588.6**	**$1,015.9**	**$766.3**	**$1,000.0**	**$878.4**	**$719.3**	**$512.4**	**$493.3**	**$468.0**	**$469.3**	**$308.6**	**$0.0**
Gold revenue	**$M**	**$2,117.8**		**$79.6**	**$114.9**	**$101.4**	**$104.8**	**$101.4**	**$99.7**	**$107.1**	**$100.4**	**$115.6**	**$118.8**	**$101.1**	**$103.6**	**$97.3**	**$103.7**	**$110.7**	**$93.2**	**$61.2**	**$53.9**	**$59.8**	**$54.7**	**$46.5**	**$46.2**	**$52.8**	**$33.5**	**$33.6**	**$22.2**	**$0.0**
Total revenue	**$M**	**$37,969.8**		**$1,265.2**	**$1,731.2**	**$1,706.9**	**$1,921.4**	**$1,970.6**	**$1,920.0**	**$2,140.9**	**$2,157.2**	**$2,177.0**	**$2,196.5**	**$2,046.4**	**$2,064.7**	**$2,001.1**	**$1,908.5**	**$1,984.7**	**$1,681.8**	**$1,077.1**	**$820.2**	**$1,059.8**	**$933.1**	**$765.8**	**$558.6**	**$546.1**	**$501.5**	**$502.8**	**$330.7**	**$0.0**

CASHFLOW SUMMARY	UNITS	FROM 2024	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050
CAPITAL COSTS																														
S3 mining fleet	$M	**$178.3**		$178.3																										
S3 expansion	$M	**$623.9**		$429.5	$152.5	$41.8																								
SE Dome pre-strip	$M	**$236.0**		$68.6	$121.5	$45.9	$0.0	$0.0																						
Smelter expansion and upgrades	$M	**$75.0**		$47.0	$28.0																									
Process plant upgrade	$M	**$13.2**		$3.9	$8.7	$0.6																								
TSF expansion	$M	**$116.0**		$23.7	$17.9	$38.5	$23.9	$11.8																						
Other site infrastructure and ancillary equipment	$M	**$27.8**		$9.8	$6.5	$3.9	$4.0	$3.5																						
TSF 3	$M	**$94.0**						$25.0	$69.0																					
Other stripping	$M	**$182.9**		$0.0	$0.0	$61.5	$79.0	$42.4																						
Dewatering Shaft & Associated	$M	**$21.6**		$21.6																										
New Trolley Line Installation	$M	**$30.7**		$11.6	$10.9	$3.8	$4.3																							
Second primary crusher	$M	**$70.0**			$5.0	$20.0	$45.0																							
Gravity gold	$M	**$0.3**		$0.3																										
Other (aggregate crusher, HPL oxygen upgrade, and other)	$M	**$18.5**		$17.5	$1.0																									
Closure costs	$M	**$128.2**																		**$21.6**	**$10.7**	**$10.7**	**$10.7**	$10.7	$10.7	$10.7	$10.7	$10.7	$10.7	$10.7
Total capital costs	**$M**	**1,816.2**		**$811.9**	**$352.0**	**$216.2**	**$156.2**	**$82.7**	**$69.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$21.6**	**$10.7**	**$10.7**	**$10.7**	**$10.7**	**$10.7**	**$10.7**	**$10.7**	**$10.7**	**$10.7**	**$10.7**
SUSTAINING COSTS																														
Mining sustaining	$M	**$1,694.2**		$77.0	$151.4	$168.7	$141.7	$96.8	$42.3	$42.2	$42.7	$42.7	$42.6	$137.4	$137.4	$137.4	$136.8	$134.8	$32.0	$37.7	$26.1	$24.3	$18.7	$12.7	$5.7	$5.0				
Process sustaining	$M	**$248.3**		$2.9	$2.2	$0.9	$1.2	$1.1	$16.1	$16.1	$16.1	$16.8	$16.8	$16.8	$16.8	$16.8	$16.8	$16.7	$16.7	$8.2	$8.2	$8.2	$8.2	$8.2	$8.2	$8.2				
Smelter sustaining	$M	**$230.7**		$9.1	$25.0	$1.8	$2.5	$0.9	$32.3	$2.3	$2.3	$2.3	$32.3	$2.3	$2.3	$2.3	$32.3	$2.3	$2.3	$2.3	$32.3	$2.3	$2.3	$2.3	$32.3	$2.3				
Other sustaining	$M	**$146.4**		$66.3	$22.1	$17.8	$16.0	$16.0	$2.1	$0.1	$0.1	$0.5		$2.1	$0.1	$0.1	$0.5		$2.1	$0.1	$0.1	$0.5		$0.1	$0.0	$0.0				
Total sustaining costs	**$M**	**$2,319.6**		**$155.3**	**$200.8**	**$189.2**	**$161.4**	**$114.7**	**$92.7**	**$60.7**	**$61.1**	**$62.3**	**$91.7**	**$158.6**	**$156.6**	**$156.6**	**$186.4**	**$153.8**	**$53.0**	**$48.4**	**$66.7**	**$35.3**	**$29.2**	**$23.4**	**$46.2**	**$15.5**	**$0.0**	**$0.0**	**$0.0**	**$0.0**
OPERATING COSTS																														
Mining																														
Ore mined to direct feed and to s/pile	$M	**$2,157.0**		$83.3	$125.1	$113.0	$102.3	$118.1	$118.8	$133.6	$115.1	$109.4	$123.8	$130.0	$126.2	$106.9	$92.6	$101.6	$78.0	$67.5	$38.7	$56.8	$78.8	$63.8	$36.8	$36.8	$0.0	$0.0	$0.0	$0.0
Ore reclaimed from stockpile	$M	**$796.4**		$43.4	$44.2	$56.2	$39.6	$37.8	$30.7	$9.6	$10.2	$12.1	$7.6	$10.4	$14.1	$15.9	$26.6	$18.6	$41.9	$0.0	$25.5	$10.0	$1.1	$22.0	$53.0	$56.7	$74.4	$74.1	$60.6	$0.0
Waste mined to dump	$M	**$6,478.3**		$294.7	$312.1	$305.9	$344.5	$331.5	$329.7	$326.7	$349.9	$348.7	$344.1	$346.6	$343.1	$341.6	$351.2	$334.8	$295.9	$342.6	$264.4	$236.5	$173.1	$111.3	$29.3	$20.3	$0.0	$0.0	$0.0	$0.0
Stockpile relocation	$M	**$64.8**		$29.7	$34.3	$0.8	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Deduct SED pre-strip and stripping	$M	**-$418.9**		-$68.6	-$121.5	-$107.4	-$79.0	-$42.4																						
Subtotal mining costs	**$M**	**$9,077.6**		**$382.5**	**$394.2**	**$368.5**	**$407.4**	**$445.1**	**$479.1**	**$469.9**	**$475.1**	**$470.2**	**$475.5**	**$487.0**	**$483.4**	**$464.4**	**$470.5**	**$455.0**	**$415.9**	**$410.1**	**$328.6**	**$303.3**	**$253.0**	**$197.1**	**$119.1**	**$113.7**	**$74.4**	**$74.1**	**$60.6**	**$0.0**
Processing and G&A																														
Oxide feed	$M	**$1,190.1**		$100.1	$75.7	$71.4	$72.1	$72.7	$73.4	$73.0	$73.3	$72.5	$72.4	$71.9	$71.8	$72.3	$71.9	$72.5	$73.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Mixed feed	$M	**$1,020.5**		$79.7	$58.4	$56.8	$75.6	$56.2	$55.9	$55.9	$55.9	$55.6	$55.5	$55.5	$55.5	$55.6	$55.5	$96.6	$96.6	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
S2 feed	$M	**$1,462.1**		$135.1	$107.2	$96.4	$79.3	$95.1	$94.5	$94.5	$94.5	$94.1	$93.9	$93.9	$93.9	$94.1	$93.9	$50.9	$50.9	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
S3 feed	$M	**$4,586.4**		$0.0	$88.3	$183.6	$173.9	$175.7	$175.7	$175.7	$175.7	$190.8	$190.4	$190.4	$190.4	$190.8	$190.4	$190.5	$190.5	$207.0	$197.1	$197.1	$197.1	$197.5	$197.1	$197.1	$197.1	$197.5	$129.1	$0.0
G&A	$M	**$1,708.5**		$51.5	$69.0	$75.5	$76.0	$76.8	$76.0	$76.0	$76.0	$76.5	$76.5	$76.5	$76.5	$76.5	$76.5	$76.4	$76.4	$57.7	$54.5	$51.2	$51.2	$51.2	$51.2	$51.2	$51.2	$51.2	$49.3	$0.0
Subtotal processing costs	**$M**	**$9,967.6**		**$366.5**	**$398.7**	**$483.6**	**$476.9**	**$476.4**	**$475.4**	**$475.1**	**$475.4**	**$489.5**	**$488.6**	**$488.1**	**$488.0**	**$489.3**	**$488.1**	**$486.9**	**$487.4**	**$264.8**	**$251.6**	**$248.3**	**$248.3**	**$248.7**	**$248.3**	**$248.3**	**$248.3**	**$248.7**	**$178.5**	**$0.0**
Total operating costs	**$M**	**$19,045.2**		**$749.0**	**$793.0**	**$852.0**	**$884.3**	**$921.5**	**$954.5**	**$945.0**	**$950.5**	**$959.6**	**$964.1**	**$975.1**	**$971.3**	**$953.7**	**$958.6**	**$941.9**	**$903.3**	**$674.8**	**$580.2**	**$551.6**	**$501.3**	**$445.8**	**$367.4**	**$362.0**	**$322.7**	**$322.8**	**$239.1**	**$0.0**
METAL COSTS (TCRCs AND ROYALTIES)																														
Cathode costs	$M	**$175.7**		$13.4	$11.7	$11.4	$10.5	$10.7	$14.1	$12.1	$13.9	$10.5	$10.7	$7.7	$7.1	$9.4	$7.8	$11.1	$13.7	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
KCS smelting/refining costs	$M	**$3,219.9**		$104.7	$146.5	$141.9	$163.2	$164.8	$159.3	$181.4	$180.1	$184.3	$188.9	$176.3	$178.5	$170.3	$165.6	$166.8	$135.7	$93.2	$71.4	$91.7	$80.6	$66.0	$47.8	$45.3	$42.9	$43.1	$29.7	$0.0
Gold refining costs	$M	**$5.9**		$0.2	$0.3	$0.3	$0.3	$0.3	$0.3	$0.3	$0.3	$0.3	$0.3	$0.3	$0.3	$0.3	$0.3	$0.3	$0.3	$0.2	$0.1	$0.2	$0.2	$0.1	$0.1	$0.1	$0.1	$0.1	$0.1	$0.0
Copper royalty (net)	$M	**$1,908.9**		$64.5	$78.2	$77.7	$87.9	$90.4	$88.1	$98.4	$99.5	$99.7	$100.5	$94.1	$95.2	$115.1	$109.1	$113.3	$96.1	$61.4	$46.3	$60.5	$53.1	$43.5	$31.0	$29.8	$28.3	$28.4	$18.7	$0.0
Gold royalty (net)	$M	**$112.6**		$4.3	$5.5	$4.9	$5.0	$4.9	$4.8	$5.1	$4.8	$5.6	$5.7	$4.9	$5.0	$5.8	$6.2	$6.6	$5.6	$3.7	$3.2	$3.6	$3.3	$2.8	$2.8	$3.2	$2.0	$2.0	$1.3	$0.0
ZCCM royalty	$M	**$1,177.1**		$39.2	$53.7	$52.9	$59.6	$61.1	$59.5	$66.4	$66.9	$67.5	$68.1	$63.4	$64.0	$62.0	$59.2	$61.5	$52.1	$33.4	$25.4	$32.9	$28.9	$23.7	$17.3	$16.9	$15.5	$15.6	$10.3	$0.0
Total metal costs	**$M**	**$6,600.0**		**$226.3**	**$295.9**	**$289.0**	**$326.5**	**$332.1**	**$326.1**	**$363.7**	**$365.4**	**$367.9**	**$374.2**	**$346.6**	**$350.2**	**$363.0**	**$348.2**	**$359.7**	**$303.5**	**$191.9**	**$146.6**	**$188.8**	**$166.1**	**$136.2**	**$99.0**	**$95.3**	**$88.9**	**$89.1**	**$60.0**	**$0.0**
MINERAL RESERVE CASHFLOW (AT $3.50/lb Cu, $1805/oz Au); PRE TAX, FROM 2024																														
Undiscounted cashflow	$M	**$8,188.8**		**-$677.3**	**$89.6**	**$160.6**	**$393.0**	**$519.6**	**$477.6**	**$771.6**	**$780.2**	**$787.1**	**$766.4**	**$566.1**	**$586.6**	**$527.7**	**$415.2**	**$529.3**	**$422.0**	**$140.4**	**$16.1**	**$273.5**	**$225.9**	**$149.8**	**$35.3**	**$62.6**	**$79.3**	**$80.2**	**$21.0**	**-$10.7**
Cumulative cashflow	$M	**$8,188.8**		**-$677.3**	**-$587.7**	**-$427.1**	**-$34.1**	**$485.4**	**$963.1**	**$1,734.7**	**$2,514.9**	**$3,302.0**	**$4,068.4**	**$4,634.6**	**$5,221.1**	**$5,748.9**	**$6,164.1**	**$6,693.3**	**$7,115.4**	**$7,255.8**	**$7,271.9**	**$7,545.3**	**$7,771.2**	**$7,921.1**	**$7,956.4**	**$8,019.0**	**$8,098.2**	**$8,178.5**	**$8,199.5**	**$8,188.8**
NPV_{10} (indicative)	$M	**$2,851.9**																												
NPV_{8} (indicative)	**$M**	$3,468.8																												
IRR	%	**46%**																												
Payback year	Year	**2028**																												
MINERAL RESERVE CASHFLOW (AT $3.50/lb Cu, $1805/oz Au); POST TAX, FROM 2024																														
Taxed paid	$M	**-$1,916.3**		-$5.3	-$11.8	-$13.1	-$12.0	-$26.8	-$37.4	-$95.9	-$155.4	-$167.6	-$173.1	-$159.6	-$144.5	-$171.9	-$147.8	-$142.0	-$124.5	-$60.8	-$12.5	-$33.9	-$65.7	-$52.4	-$27.6	-$15.4	-$17.8	-$20.0	-$14.2	-$7.4
Undiscounted cashflow	$M	**$6,272.5**		**-$682.6**	**$77.8**	**$147.5**	**$381.0**	**$492.8**	**$440.2**	**$675.7**	**$624.8**	**$619.5**	**$593.3**	**$406.5**	**$442.1**	**$355.9**	**$267.4**	**$387.3**	**$297.6**	**$79.6**	**$3.6**	**$239.6**	**$160.2**	**$97.5**	**$7.7**	**$47.2**	**$61.5**	**$60.2**	**$6.8**	**-$18.0**
Cumulative cashflow	$M	**$6,272.5**		**-$682.6**	**-$604.8**	**-$457.3**	**-$76.3**	**$416.5**	**$856.7**	**$1,532.4**	**$2,157.3**	**$2,776.8**	**$3,370.1**	**$3,776.6**	**$4,218.7**	**$4,574.6**	**$4,842.0**	**$5,229.2**	**$5,526.8**	**$5,606.4**	**$5,610.0**	**$5,849.6**	**$6,009.8**	**$6,107.3**	**$6,115.0**	**$6,162.1**	**$6,223.6**	**$6,283.8**	**$6,290.6**	**$6,272.5**
NPV_{10} (indicative)	$M	**$2,223.1**																												
NPV_{8} (indicative)	**$M**	$2,700.2																												
IRR	%	**42%**																												
Payback year	Year	**2028**																												

CASHFLOW SUMMARY	UNITS	FROM 2024	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050
GOLD ADDITIONAL TO MINERAL RESERVE ANODE CONTENT																														
total additional gold	**k(t)oz**	**929.6**		**27.6**	**28.4**	**55.6**	**61.8**	**64.0**	**43.3**	**53.6**	**50.2**	**53.8**	**55.7**	**45.7**	**47.3**	**41.8**	**42.8**	**53.3**	**35.1**	**31.4**	**20.7**	**30.4**	**24.3**	**16.8**	**11.4**	**12.2**	**8.0**	**7.9**	**6.4**	**0.0**
ADDITIONAL GROSS REVENUE																														
Average gold price	$/(t)oz	**$1,805**		$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$1,805	$0
additional gold revenue	**$M**	**$1,677.9**		**$49.9**	**$51.2**	**$100.3**	**$111.6**	**$115.6**	**$78.2**	**$96.7**	**$90.7**	**$97.2**	**$100.6**	**$82.6**	**$85.4**	**$75.4**	**$77.2**	**$96.3**	**$63.3**	**$56.7**	**$37.3**	**$54.9**	**$43.9**	**$30.3**	**$20.6**	**$22.0**	**$14.4**	**$14.3**	**$11.5**	**$0.0**
ADDITIONAL OPERATING COSTS																														
additional operating cost	**$M**	**$90.1**		**$4.9**	**$4.1**	**$4.2**	**$4.2**	**$4.3**	**$3.9**	**$4.0**	**$3.9**	**$4.1**	**$4.1**	**$3.8**	**$3.9**	**$3.8**	**$3.9**	**$4.2**	**$3.8**	**$2.9**	**$2.7**	**$2.9**	**$2.8**	**$2.6**	**$2.6**	**$2.7**	**$2.4**	**$2.4**	**$1.2**	**$0.0**
ADDITIONAL ROYALTY																														
Gold royalty at 6%	$M	**$100.7**		$3.0	$3.1	$6.0	$6.7	$6.9	$4.7	$5.8	$5.4	$5.8	$6.0	$5.0	$5.1	$4.5	$4.6	$5.8	$3.8	$3.4	$2.2	$3.3	$2.6	$1.8	$1.2	$1.3	$0.9	$0.9	$0.7	$0.0
ZCCM royalty at 3.1%	$M	**$52.0**		$1.5	$1.6	$3.1	$3.5	$3.6	$2.4	$3.0	$2.8	$3.0	$3.1	$2.6	$2.6	$2.3	$2.4	$3.0	$2.0	$1.8	$1.2	$1.7	$1.4	$0.9	$0.6	$0.7	$0.4	$0.4	$0.4	$0.0
total gold royalty	**$M**	**$152.7**		**$4.5**	**$4.7**	**$9.1**	**$10.2**	**$10.5**	**$7.1**	**$8.8**	**$8.3**	**$8.8**	**$9.2**	**$7.5**	**$7.8**	**$6.9**	**$7.0**	**$8.8**	**$5.8**	**$5.2**	**$3.4**	**$5.0**	**$4.0**	**$2.8**	**$1.9**	**$2.0**	**$1.3**	**$1.3**	**$1.0**	**$0.0**
net gold royalty	**$M**	**$165.0**		**$4.2**	**$4.0**	**$7.9**	**$8.8**	**$9.1**	**$6.2**	**$7.6**	**$7.2**	**$7.7**	**$7.9**	**$6.5**	**$31.4**	**$6.9**	**$7.0**	**$8.8**	**$5.8**	**$5.2**	**$3.4**	**$5.0**	**$4.0**	**$2.8**	**$1.9**	**$2.0**	**$1.3**	**$1.3**	**$1.0**	**$0.0**
CASHFLOW INCLUSIVE OF ADDITIONAL GOLD AND ASSOCIATED COSTS; PRE-TAX, FROM 2024																														
Project cashflow	$M	**$8,188.8**		-$677.3	$89.6	$160.6	$393.0	$519.6	$477.6	$771.6	$780.2	$787.1	$766.4	$566.1	$586.6	$527.7	$415.2	$529.3	$422.0	$140.4	$16.1	$273.5	$225.9	$149.8	$35.3	$62.6	$79.3	$80.2	$21.0	-$10.7
Additional cashflow	$M	**$1,422.8**		$40.8	$43.1	$88.2	$98.5	$102.2	$68.1	$85.0	$79.6	$85.4	$88.5	$72.2	$50.1	$64.8	$66.3	$83.3	$53.8	$48.6	$31.2	$47.0	$37.1	$24.9	$16.2	$17.4	$10.7	$10.6	$9.3	$0.0
Undiscounted cashflow	**$M**	**$9,611.6**		**-$636.5**	**$132.7**	**$248.7**	**$491.5**	**$621.7**	**$545.8**	**$856.6**	**$859.8**	**$872.6**	**$854.9**	**$638.4**	**$636.6**	**$592.5**	**$481.5**	**$612.6**	**$475.8**	**$189.1**	**$47.2**	**$320.5**	**$263.0**	**$174.8**	**$51.5**	**$79.9**	**$89.9**	**$90.9**	**$30.3**	**-$10.7**
Cumulative cashflow	**$M**	**$9,611.6**		**-$636.5**	**-$503.8**	**-$255.1**	**$236.4**	**$858.1**	**$1,403.9**	**$2,260.6**	**$3,120.4**	**$3,992.9**	**$4,847.8**	**$5,486.2**	**$6,122.8**	**$6,715.3**	**$7,196.8**	**$7,809.4**	**$8,285.3**	**$8,474.3**	**$8,521.5**	**$8,842.0**	**$9,105.0**	**$9,279.8**	**$9,331.3**	**$9,411.2**	**$9,501.2**	**$9,592.0**	**$9,622.3**	**$9,611.6**
NPV_{10} (indicative)	**$M**	**$3,458.8**																												
NPV_{8} (indicative)	**$M**	**$4,169.1**																												
IRR	%	**56%**																												
Payback year	Year	**2027**																												
CASHFLOW INCLUSIVE OF ADDITIONAL GOLD AND ASSOCIATED COSTS; POST-TAX, FROM 2024																														
Taxes paid	$M	**-$2,322.0**		-$10.6	-$21.5	-$26.2	-$33.4	-$51.9	-$57.0	-$113.5	-$174.3	-$186.6	-$193.1	-$178.1	-$199.6	-$192.0	-$166.9	-$163.8	-$144.4	-$75.7	-$21.9	-$45.3	-$80.1	-$61.4	-$33.6	-$20.3	-$21.9	-$23.1	-$17.1	-$8.7
Undiscounted cashflow	**$M**	**$7,289.7**		**-$647.1**	**$111.2**	**$222.6**	**$458.1**	**$569.9**	**$488.8**	**$743.2**	**$685.5**	**$686.0**	**$661.8**	**$460.2**	**$437.0**	**$400.5**	**$314.6**	**$448.8**	**$331.4**	**$113.4**	**$25.3**	**$275.2**	**$182.9**	**$113.4**	**$17.9**	**$59.6**	**$68.1**	**$67.7**	**$13.2**	**-$19.4**
Cumulative cashflow	**$M**	**$7,289.7**		**-$647.1**	**-$535.9**	**-$313.3**	**$144.8**	**$714.7**	**$1,203.5**	**$1,946.6**	**$2,632.1**	**$3,318.1**	**$3,979.9**	**$4,440.1**	**$4,877.2**	**$5,277.6**	**$5,592.2**	**$6,041.0**	**$6,372.4**	**$6,485.8**	**$6,511.1**	**$6,786.3**	**$6,969.2**	**$7,082.6**	**$7,100.4**	**$7,160.1**	**$7,228.2**	**$7,295.9**	**$7,309.1**	**$7,289.7**
NPV_{10} (indicative)	**$M**	**$2,675.8**																												
NPV_{8} (indicative)	**$M**	**$3,218.6**																												
IRR	%	**50%**																												
Payback year	Year	**2027**																												

Table 22-3 Mineral Reserve cashflow model, pre-tax and post-tax summaries

Pre-tax models (from 2024)	Long term metal pricing ($/lb Cu)	($/oz Au)	Undiscounted Cashflow ($M)	NPV_{10} ($M)	IRR (%)
Base Case (Mineral Reserve)	**$3.50**	$1,805	$8,188.8	$2,851.9	46%
Inc. additional gold	**$3.50**	$1,805	$9,611.6	$3,458.8	56%
Higher Cu price	**$4.02**	$1,805	$13,348.5	$5,179.8	106%
Inc. additional gold	**$4.02**	$1,805	$14,807.6	$5,810.6	128%
Post-tax models (from 2024)	**Long term metal pricing ($/lb Cu)**	**($/oz Au)**	**Undiscounted Cashflow ($M)**	**NPV_{10} ($M)**	**IRR (%)**
Base Case (Mineral Reserve)	**$3.50**	$1,805	$6,272.5	$2,223.1	42%
Inc. additional gold	**$3.50**	$1,805	$7,289.7	$2,675.8	50%
Higher Cu price	**$4.02**	$1,805	$9,943.1	$3,951.2	89%
Inc. additional gold	**$4.02**	$1,805	$10,982.4	$4,416.2	104%

Table 22-4 Mineral Reserve cashflow model sensitivity analysis, pre-tax and before VAT offsets

Variable	Undiscounted Cashflow ($M)	Delta (%)	NPV_{10} ($M)	Delta (%)
Base Case	**$7,915.2**		**$2,705.0**	
Recovery +10%	**$11,172.4**	141%	**$4,051.4**	150%
Mining costs -20%	**$9,730.7**	123%	**$3,445.9**	127%
Processing costs -20%	**$9,567.0**	121%	**$3,362.8**	124%
Inc. additional gold	**$9,350.3**	118%	**$3,313.1**	122%
Mining costs -10%	**$8,822.9**	111%	**$3,075.4**	114%
Processing costs -10%	**$8,741.1**	110%	**$3,033.9**	112%
Metal costs -20%	**$8,595.5**	109%	**$2,988.1**	110%
Capex costs -20%	**$8,278.4**	105%	**$2,986.1**	110%
Metal costs -10%	**$8,255.3**	104%	**$2,846.6**	105%
Capex costs -10%	**$8,096.8**	102%	**$2,845.6**	105%
Capex costs +10%	**$7,733.5**	98%	**$2,564.4**	95%
Metal costs +10%	**$7,575.0**	96%	**$2,563.4**	95%
Capex costs +20%	**$7,551.9**	95%	**$2,423.9**	90%
Metal costs +20%	**$7,234.9**	91%	**$2,421.8**	90%
Processing costs +10%	**$7,089.3**	90%	**$2,376.1**	88%
Mining costs +10%	**$7,007.4**	89%	**$2,334.6**	86%
Processing costs +20%	**$6,263.3**	79%	**$2,047.2**	76%
Mining costs +20%	**$6,099.6**	77%	**$1,964.1**	73%
Recovery -10%	**$4,657.9**	59%	**$1,358.6**	50%

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 Mineral Resource estimates for Kansanshi North West, Main and South East were supported by extensive drill hole coverage. Open pit mining has exposed the prevailing deposit geology at the North West and Main deposits, providing valuable support for the estimates that under-pin geology and mineralisation models. QAQC of samples confirmed representative sample assay results from diamond drilling and from RC drilling. The geological understanding and resulting models align well with the extensive deposit geology exposed in current pits. The large number of close spaced RC holes enhance confidence in local geology and grade continuity within the respective domains of mineralisation. The Qualified Person, Carmelo Gomez, considers the 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, with verification of data by FQM, plus 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 complies with the reporting requirements of NI 43-101.

25.1.2 Database validation

FQM has employed dedicated database administrators to ensure data integrity. The SQL database, hosting the data, has built-in validations and constraints. The database export provided for Mineral Resource estimation did no encounter any significant issues.

25.1.3 Data

The dataset used in the Mineral Resource estimate is substantial, comprising data from 2,220 exploration diamond drill holes and 75,203 RC grade control holes. Currently, the spacing of diamond drilling is insufficient to adequately define continuity of veins, supporting the inclusion of data from RC grade control holes.

25.1.4 Geological logging

Geological logging data appears to be of good quality. Criteria for defining material as vein were established based on geological logging and vein percentage. Material was defined as vein if LITH1 was logged as vein or if Vein_Pct > 50%. This was considered adequate to define the vein mineralisation.

Additional logging criteria were defined for reverse circulation close-spaced drilling grid to improve the identification.

25.1.5 Geological model

Significant improvements have been made to the geological model since the previous estimate, particularly regarding vein wireframes and estimation methods. Reconciliation between DD only vein estimates and close spaced RC drilled vein estimates are now within 5%.

Strata-associated mineralisation has also been better constrained with improvements in the categorical indicator estimation method using dynamic anisotropy and tighter sample selection routines.

Weathering surfaces delineated saprolite, saprock and fresh material. Both weathering and oxidation volumes have been improved via a more thorough categorical indicator estimation method. This method provides a more realistic estimate of the volumes of weathering and oxidation.

25.1.6 Bulk density and gangue acid consumption

Additional data collected since 2020 lead to the revision of bulk density and GAC values for different units, and enabling spatial estimates, resulting in improved confidence in the tonnage of the Mineral Resource.

25.1.7 Assay QAQC

The QAQC programme implemented at Kansanshi for the exploration samples was rigorous and comprehensive. Sufficient QAQC material has been included with the drill samples to give confidence that the assay results obtained accurately reflect sample grades. Ongoing monitoring occurs with monthly QAQC reports generated, and any issues were investigated in a timely manner.

25.1.8 Mineral Resource estimate

The grade estimate was completed in three stages:

1. The stratigraphic mineralisation was estimated using ordinary kriging for close-spaced and wide-spaced drilled areas separately. Wide-spaced drilled areas were post processed with localised uniform conditioning. Hard boundaries were used between stratigraphic units.
2. Veins defined by 3D wireframes were estimated using ordinary kriging.
3. Veins with insufficient drill hole support (wide grid spacings – SMUDRSCL 0) were estimated using a categorical indicator followed by a true width ordinary kriged estimate and an ordinary kriged grade estimate. Hard boundaries were used between oxide-mixed and fresh sulphide units.

Block grade estimates were validated, combined and classified. Classification was guided by confidence in the geology model and estimation methodology which informs volume, drill hole spacing, QAQC, and confidence in the grade estimate which was informed by kriging efficiency, slope of regression, search pass, drilling and sample density, and guided by an economic optimised pit shell for reasonable potential for economical extraction. Laterite and refractory mineralisation were not classified.

The updated Mineral Resource estimate shows a 43% increase in Measured and Indicated tonnes with a 6% decrease in TCu grade compared to the 2024 Mineral Resource estimate. Inferred resources reduced by 70% due to upgrades into the Measured and Indicated categories.

Reconciliation from historical production data was used to support and validate the Mineral Resource estimate. Results suggest that plant claimed metal is 5% lower than Mineral Resource estimated metal. This magnitude of risk is aligned with the assigned Mineral Resource classification.

A regularised 10 m by 10 m by 5 m block model was provided for Mineral Reserve conversion.

25.2 Mine planning and Mineral Reserve estimation

The Mineral Reserve estimate for Kansanshi is the product of a thorough and conventional process reflecting detailed ultimate pit design guided by a selected design pit shell, reflective of a mature operating mine, conformable with the existing extents of open pit mining, and with the practical cutbacks required to achieve the proposed production expansion.

The mine planning process incorporates the best available information, including latest geotechnical information and updated mining, processing and metal costs. A detailed production schedule has been prepared for the Kansanshi life of mine, along with a waste dumping plan which allows for partial in-pit backfilling of the depleted North West Pit. The basis of a reported uplift in the Mineral Reserve inventory is a fundamental update to the Mineral Resource estimate.

Worthy of note is that the production scheduling aspects of the Mineral Reserve estimation process have considered potential mining recovery losses more thoroughly than was previously the case for the 2020 and earlier estimates (FQM, May 2015, September 2020). A mine planning/modelling process has been further developed to emulate the variable extents of planned mining recovery losses that will be incurred in the future, when marking-out ore polygons to suit varying mineralisation styles in all pits, including South East Dome. Importantly, the adopted methodology has been aligned with the shorter-term projections within the mine budgeting timeframe.

The following information relates to risks and uncertainties around the Mineral Reserve estimate.

25.2.1 Mining and primary equipment

There is considered to be minimal risk attributable to the mining method and to the primary equipment in use at Kansanshi and as proposed for future mining. The method and equipment are conventional and suitable for the scale of the expanding mining operations.

With the S3 expansion and as the mine transitions towards a bulk mining operation with longer term higher annual stripping ratios, certain operational improvements will be required to facilitate the projected annual material movement requirements. High and sustained haul truck productivity will be necessary and to this end, programmes are in place to reduce non-productive truck cycle times and to improve equipment maintenance practices. Additional trolley-assisted haulage routes are planned, extending to flat segments on surface to cater for long hauls to waste dumps and from reclaim stockpiles. Productivity enhancements are also proposed via more efficient truck dispatching, and also through optimising drill and blast patterns.

25.2.2 Stockpile reclamation

An important aspect of the Mineral Reserves plan is the early reclamation of existing sulphide stockpiles, between 2025 and 2029, to supplement S3 plant feed and to free-up waste dumping space on surface. During this five year period, approximately 60% reducing to 35% of the total combined plant feed will come from stockpile reclaim. There is some uncertainty in the scheduled grade of ore reclaimed from these stockpiles and hence, a downgrade factor has been applied during the production scheduling process (i.e. emulating potential ore loss).

25.2.3 Waste handling

As already noted, a significant aspect of the Mineral Reserve production scheduling is the spectre of high strip ratio cutbacks to the ultimate pit perimeter. These have been dealt with in the schedule by deferring these cutbacks to the final years of mine life. Much of this waste is near-surface overburden and hence the schedule has been devised to, as far as possible, distribute these volumes such that they could be bulk mined by a dedicated fleet.

25.2.4 Geotechnical risks

Previous technical reports have made reference to geotechnical risks to mine planning and the need for continuous review and update of pit design parameters. This latest mine design and plan has been cognisant of information and geotechnical recommendations arising from improved definition of weathering extents and domains that have been incorporated into the latest Mineral Resource model database.

25.2.5 Processing recovery projections

Another aspect previously mentioned as having a bearing on the Mineral Reserve estimate, and particularly the recovered metal associated with the reserve inventory, is the use of variable process recovery relationships. These relationships continue to be reviewed and refined for the purposes of five year planning/budgeting and for longer term projections. These refinements will need to include South East Dome projections, which in time will benefit from actual feed performance findings as opposed to testwork findings.

25.2.6 Water management

Work is in-hand on the re-establishment of a portal and lateral underground development for proposed new pit dewatering infrastructure at Kansanshi. This is in relation to the existing dewatering shaft and its limitations in serving the proposed expanded and deepened Main Pit.

25.2.7 Mining licence, environmental and social

To the Company's knowledge, the Kansanshi Project is not considered by any applicable environmental regulatory authority to be a risk to the environment, and thereby jeopardising the continued development of the Project.

25.3 Processing

The processing facilities have been expanded and improved over many years, and the flexibility of the current circuits allows ore types to be treated through different process routes.

Variable process recovery relationships have been developed for all copper ore types, based on historic operating data. These are being continuously updated, and provide confidence in the recovery numbers used in the production schedule, Mineral Reserve estimate and the financial model for Kansanshi.

Sulphuric acid produced by the smelter has provided the opportunity to leach mixed ore flotation tails, leading to increased recoveries from these ore types. A new flotation cleaner circuit has been installed, and ongoing reagent optimisations are aimed at increasing copper grades and reducing carbon levels in concentrates, leading to increases in smelter throughput.

Designs for the S3 expansion have built on experiences gained from the current operations at Kansanshi, and at the Company's Sentinel and Cobre Panama Projects. Consequently, there is considered to be minimal technical risk for the S3 Project.

The anticipated increase in concentrate generation due to the S3 expansion has been adequately addressed in designs for expanding the cleaner flotation circuit installed in 2022, and for increased throughput at the smelter and through the high pressure leach circuit.

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 estimation recommendations

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

1. Continue testing and developing geology modelling methods to better constrain mineralisation.
2. Maintain ongoing structural and geological field mapping.
3. Conduct additional diamond drilling to define the limits of mineralisation, particularly at the South East deposit.
4. Investigate the potential value of sequential copper leach analysis in non-fresh areas.
5. Explore data acquisition techniques to better support the definition of vein volumes.
6. Improve reconciliation information with additional measuring points to increase data accuracy and improve the understanding on estimation variances for each mineralisation style.

26.2 Mineral Reserve estimation recommendations

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

1. A methodology has been adopted for this latest Mineral Reserve which provides an estimate of potential ore losses incurred through planned and unplanned events within the mining process. The methodology incorporates elements of modelling, risk consideration, reconciliation against production records and alignment against short-term budgeting projections. This methodology should continue to be reviewed, refined and improved, especially in regard to the veracity of production tracking data.
2. Given the variability of ore and waste handling requirements, in circumstances of differing ore haulage destinations, expansive surface waste dumping and backfill locations, plus numerous stockpile reclaim sites, the modelling of ore and waste haulage profiles should continue to be an important component of mining cost estimation.
3. Aspects of the mine planning process that ought to be continuously reviewed and updated include the geotechnical design recommendations (also accounting for slope depressurisation recommendations) and the specification of varying process recovery relationships.

26.3 Mineral Processing recommendations

Part of the feed material for the S3 circuit will be from the South East Dome pit, mining of which has only recently commenced. Some samples from this deposit have been collected for mineralogical examination, and have been subject to limited metallurgical testwork.

Additional testwork was conducted in 2023 to better define the comminution characteristics of the South East Dome plant feed. This work is continuing in 2024 with the aim of understanding the link between hardness parameters and the lithology and alteration characteristics of the plant feed from this pit.

ITEM 27 REFERENCES

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ITEM 28 CERTIFICATES

Carmelo Gomez Dominguez
First Quantum Minerals Ltd
18–32 Parliament Place, West Perth, Western Australia, 6005
Tel +61 8 9346 0100; carmelo.gomez@fqml.com

I, Carmelo Gomez Dominguez, do hereby certify that:

1. I am a Group Principal Geologist, Mine and Resources, 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 December 2023 (the "Technical Report").
3. I am a professional geologist having graduated with a Bachelor of Science degree with Honours (2003) in Geology from Huelva University, Spain.
4. I am a Fellow of the European Federation of Geologists (EFG).
5. I have worked as a geologist for a total of twenty one years since my graduation from university. I have gained over 14 years' experience in production geology. During the last ten years I have consulted to and held senior technical Mineral Resource positions in copper mining companies operating in Europe, 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 have worked at Kansanshi for three years from 2018 to 2021, and most recently personally inspected the Kansanshi property described in the Technical Report in June 2022, for a duration of 20 days.
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 7-12 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 to be disclosed and to make the Technical Report not misleading.

Signed and dated this 23rd day of July 2024 at West Perth, Western Australia, Australia.

Carmelo Gomez Dominguez

Michael Lawlor
First Quantum Minerals Ltd
18-32 Parliament Place, West Perth, Western Australia, 6005
Tel +61 8 9346 0100; mike.lawlor@fqml.com

I, Michael Lawlor, do hereby certify that:

1. I am a Mining Technical Advisor 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 December 2023 (the "Technical Report").
3. I am a professional mining engineer having graduated with an undergraduate degree of Bachelor of Engineering (Honours) from the Western Australian School of Mines in 1986. In addition, I have obtained a Master of Engineering Science degree from the James Cook University of North Queensland (1993), and subsequent Graduate Certificates in Mineral Economics and Project Management from Curtin University (Western Australia).
4. I am a Fellow of the Australasian Institute of Mining and Metallurgy.
5. I have worked as a mining and geotechnical engineer for a period in excess of thirty five years since my graduation from university. Within the last fifteen 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 March 2023, for a duration of five days.
8. I am responsible for the preparation of those portions of the Technical Report relating to Mineral Reserve estimation and Mining, namely Items 15 and 16, respectively, and for Items 1, 2, and 18 to 26.
9. I am not independent (as defined by Section 1.5 of NI 43-101) of First Quantum Minerals Ltd.
10. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement has been in mine planning and the preparation of long term mining plans and production schedules, commencing in 2013.
11. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.
12. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed and to make the Technical Report not misleading.

Signed and dated this 23rd day of July 2024 at West Perth, Western Australia, Australia.

M. Lawlor

Michael Lawlor

Andrew Briggs
First Quantum Minerals Ltd
18-32 Parliament Place, West Perth, Western Australia, 6005
Tel +61 8 9346 0100; andy.briggs@fqml.com

I, Andrew Briggs, do hereby certify that:

1. I am the Group Consulting Project Metallurgist employed by First Quantum Minerals Ltd.
2. This certificate applies to the technical report entitled “Kansanshi Operations, North West Province, Zambia, NI 43-101 Technical Report” dated effective 31st December 2023 (the “Technical Report”).
3. I am a professional metallurgist having graduated in 1974 from the Imperial College (Royal School of Mines), London, with a BSc (Eng) First Class in Metallurgy.
4. I am a Fellow of the Southern African Institute of Mining and Metallurgy.
5. I have worked as a process engineer and metallurgist since graduation in 1974 (46 years); the first 13 years of which were in operating positions up to Metallurgical Manager in the gold mining industry. This was followed by 19 years in engineering companies in Process Design for projects worldwide, and finally 13 years with First Quantum Minerals Ltd as a Process Consultant.
6. I have read the definition of “qualified person” as set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a “qualified person” for the purposes of NI 43-101.
7. I most recently personally inspected the Kansanshi property described in the Technical Report for three days in July 2023.
8. I am responsible for the preparation of those portions of the Technical Report relating to mineral processing/metallurgical testing and recovery methods, namely Items 13 and 17, respectively. I am also responsible for the estimates in Item 21 pertaining to processing, plus general and administration costs.
9. I am not independent (as defined by Section 1.5 of NI 43-101) of First Quantum Minerals Ltd.
10. I have been involved with the property that is the subject of the Technical Report, since 2007. This work has included metallurgical testwork, process design for the plant and associated infrastructure, upgrades and engineering studies.
11. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.
12. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed and to make the Technical Report not misleading.

Signed and dated this 23rd day of July 2024 at West Perth, Western Australia, Australia.

Andrew Briggs

Andrew Briggs