IVANHOE MINES LTD

Kipushi Project

Kipushi 2022 Feasibility Study

February 2022

Job No. 21003

IMPORTANT NOTICE

This notice is an integral component of the Kipushi 2022 Feasibility Study (Kipushi 2022 FS) and should be read in its entirety and must accompany every copy made of the Kipushi 2022 FS. The Kipushi 2022 FS has been prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

The Kipushi 2022 FS has been prepared for Ivanhoe Mines Limited (Ivanhoe) by OreWin Pty Ltd (OreWin), The MSA Group (Pty) Ltd (MSA), SRK Consulting (South Africa) (Pty) Ltd (SRK) and METC Engineering Pty Ltd (METC). The Kipushi 2022 FS is based on information and data supplied to OreWin and MSA by Ivanhoe and other parties and where necessary the authors have assumed that the supplied data and information are accurate and complete.

The conclusions and estimates stated in the Kipushi 2022 FS are to the accuracy stated in the Kipushi 2022 FS only and rely on assumptions stated in the Kipushi 2022 FS. The results of further work may indicate that the conclusions, estimates and assumptions in the Kipushi 2022 FS need to be revised or reviewed.

OreWin has used its experience and industry expertise to produce the estimates and approximations in the Kipushi 2022 FS. Where OreWin has made those estimates and approximations, it does not warrant the accuracy of those amounts and it should also be noted that all estimates and approximations contained in the Kipushi 2022 FS will be prone to fluctuations with time and changing industry circumstances.

The Kipushi 2022 FS should be construed in light of the methodology, procedures, and techniques used to prepare the Kipushi 2022 FS. Sections or parts of the Kipushi 2022 FS should not be read or removed from their original context.

The Kipushi 2022 FS is intended to be used by Ivanhoe, subject to the terms and conditions of its contracts with OreWin, MSA, and METC. Recognising that Ivanhoe has legal and regulatory obligations, OreWin, MSA, and METC have consented to the filing of the Kipushi 2022 FS with Canadian Securities Administrators and its System for Electronic Document Analysis and Retrieval (SEDAR). Except for the purposes legislated under provincial securities laws, any other use of this report by any third party is at that party's sole risk.

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Title Page

Project Name:	Kipushi Project
Title:	Kipushi 2022 Feasibility Study
Location:	Haut-Katanga Province Democratic Republic of the Congo
Effective Date of Technical Report:	14 February 2022
Effective Date of Mineral Resources:	14 June 2018
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Effective Date of Mineral Reserves:	14 February 2022

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Signature Page

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1 SUMMARY

1.1 Introduction

The Kipushi 2022 Feasibility Study (Kipushi 2022 FS) is an independent NI 43-101 Technical Report (the Report) prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for Ivanhoe Mines Ltd. (Ivanhoe), for the Kipushi Project (the Project) located in the Democratic Republic of Congo (DRC).

For the purpose of this Report, the Project means the project covered by Exploitation Permit No. 12434. The Project is located adjacent to the town of Kipushi in the south-western part of the Haut Katanga Province in the DRC, adjacent to the border with Zambia. Kipushi town is situated approximately 30 km south-west of Lubumbashi, the capital of Haut Katanga Province. Kipushi Holding Limited (a wholly owned subsidiary of Ivanhoe Mines Ltd. (Ivanhoe)) (Kipushi Holding) and Gécamines S.A., previously known as La Générale des Carrières et Des Mines (Gécamines) have a joint venture agreement (JV Agreement) over the Kipushi Project. Kipushi Holding and Gécamines respectively own directly 68% and 32% of the Kipushi Project through Kipushi Corporation SA (KICO), the mining right holder of the Kipushi Project.

The JV Agreement, which was signed on 14 February 2007 and subsequently amended several times, established KICO for the exploration, development, production, and product marketing of the Kipushi Project.

Ivanhoe's interest in KICO was acquired in November 2011 and includes mining rights for copper, cobalt, zinc, silver, lead, and germanium as well as the underground workings and related infrastructure, inclusive of a series of vertical mine shafts.

As announced on February 14, 2022, Kipushi Holding and Gécamines have agreed to commercial terms that will form the basis of a new Kipushi joint-venture agreement in order to establish a robust framework for the mutually beneficial operation of the Kipushi Mine. The new agreement remains subject to execution of definitive documentation.

The previous Technical Report was the Kipushi 2019 Resource Update with an effective date in March 2019. Ivanhoe has undertaken further study work following the Kipushi 2019 Resource Update that has formed the basis of the Kipushi 2022 FS, which summarises the current Ivanhoe development strategy for the Kipushi Project. The Kipushi 2022 FS provides an update of the Kipushi Project Mineral Reserve, with the Mineral Resource from the Kipushi 2019 Resource Update remaining the same.

1.2 Location

The Lubumbashi region is characterised by a humid subtropical climate with warm rainy summers and mild dry winters. Most rainfall occurs during summer and early autumn (November–April) with an annual average rainfall of 1,208 mm. Average annual maximum and minimum temperatures are 28°C and 14°C respectively.

Historical mining operations at the Kipushi Project operated year-round, and it is expected that any future mining activities at the Kipushi Project would also be able to be operated on a year-round basis. KICO currently employs over 450 DRC nationals from the town of Kipushi, preparing and advancing the project development on the surface and underground in activities.

1.3 Property Description and Ownership

The Project is located adjacent to the town of Kipushi in the south-western part of the Haut-Katanga Province in the DRC, adjacent to the border with Zambia. Kipushi town is situated approximately 30 km south-west of Lubumbashi, the capital of Haut-Katanga Province.

The Kipushi Mine is a past-producing, high-grade underground zinc–copper mine in the Central African Copperbelt, which operated from 1924–1993. The mine produced approximately 60 Mt at 11.03% Zn and 6.78% Cu including, from 1956–1978, approximately 12,673 t of lead and 278 t of germanium (Ivanhoe, 2014). Mining at Kipushi began as an open pit operation but by 1926 had become an underground mine, with workings down to 1,150 below surface (mL) In 1993, the mine was put on care and maintenance due to a combination of economic and political factors.

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1.4 Mineral and Surface Rights, Royalties, and Agreements

KICO holds the exclusive right to engage in mining activities within the Kipushi Project area through a mining right, Exploitation Permit No. 12434 (PE12434), valid until 3 April 2024, and covering 07 mining squares and approximately 505 ha. This permit is renewable under the terms of the DRC Mining Code.

The Exploitation Permit No. 12434 resulted from the partial transfer of Exploitation Permit No. 481 previously held by Gécamines, was granted by Ministerial Order No. 0290/CAB.MIN/MINES/01/2011 dated 2 July 2011 and is evidenced by Exploitation Certificate No. CAMI/CE/6368/11 dated 22 July 2011, and granted KICO the exclusive right to perform exploration, development and exploitation works concerning silver, cobalt, copper, germanium, and zinc.

Exploitation Permit No. 12434 is still under a situation of Force Majeure duly approved by Decision No. CAMI/DG/FM/19/2012 dated 2 April 2012 until the Kipushi underground mine and its facilities have been refurbished. This permit can be extended following the end of the force majeure event currently impacting PE12434.

The Cadastre Minier (CAMI) challenged the continuation of the Force Majeure impacting Exploitation Permit No. 12434 in a letter received by KICO on 9 February 2021 and requested KICO to confirm, with evidence, the continuation of the approved Force Majeure. KICO confirmed the continuation of Force Majeure in a letter dated 25 February 2021, clarifying its aim of finalising critical works by 24 March 2022 in order to subsequently note the end of the Force Majeure. In order to examine this letter, CAMI requested, on 15 March 2021, the performance of an on-site survey by the Provincial Department of Mines and requested this Provincial Department of Mines to send CAMI minutes enabling CAMI to assess the accuracy of KICO's statements. After on-site inspection performed by the Provincial Department of Mines, this latter concluded notably in its minutes dated of 6 May 2021 that it considered that the facts constituting the case of Force Majeure affecting the exercise and use of the mining right relating to Exploitation Permit No. 12434 continue and that the extension of the validity period can be considered. There were no further development at CAMI's level since those minutes, to the best of Ivanhoe's knowledge so far.

The Zambian and DRC governments have both contracted FlexiCadastre (Spatial Dimension) to assist with the management of the mining rights of both states. This enables alignment regarding the management of mining rights on both sides of the border.

The boundaries of Exploitation Permit No. 12434, indicated in the Exploitation Certificate, cross the international border, as do some of the coordinates on the permit held as defined by the CAMI. DRC permits are made up of cadastral squares (carrés) meaning the coordinates of the permit boundary (defined to the international border) and the permit blocks (defined by the cadastral squares) may not be coincidental.

As the DRC Mining Code does not apply in Zambia and therefore has no jurisdiction in Zambia, the right for KICO to mine stops at the international border, and any part of the exploitation permit area extending beyond the DRC borders are excluded from the exploitation permit.

The mineralisation at the Kipushi Project may extend, at depth, beyond the DRC border into Zambia. KICO does not have an agreement with the Zambian government which would permit it to explore for or exploit any Mineral Resources that may be in Zambia. The current Mineral Resource estimates presented for the Kipushi Project only make reference to those Mineral Resources which lie within the DRC.

On 9 March 2018, Law No. 18/001 amending the 2002 Mining Code was promulgated, followed by the adoption of Decree No. 18/024 of 8 June 2018 that amended and completed the 2003 Mining Regulations.

For the purpose of this report, the economic analysis is based on the 2018 Mining Code. According to the 2018 Mining Code, a company holding an exploitation permit is subject to payment of mining royalties. The royalty is due upon the sale of the product and is calculated at 3.5% of the value of metal sold, which is assumed to be 65% for zinc concentrate (51–60% Zn content).

Holders of mining rights are, generally, subject to taxes, customs, and levies defined in the 2018 Mining Code for all mining activities carried out by the holder in the DRC. Nothing in this document nor any payment by KICO, under duress and in order to avoid sanctions, made pursuant to the 2018 Mining Code and Regulations or subsequent laws and regulations could be interpreted as constituting a waiver to any stability guarantee that would benefit to KICO, Kipushi Holding Limited or Ivanhoe or something equivalent.

KICO will continue to monitor the regulatory provisions to be adopted, ensuring as far as possible, continued adequate adherence to the relevant legislative requirements. Detailed discussions are ongoing with the aim of resolving, in a fair and equitable manner, the mining industry's concerns with the 2018 Mining Code.

1.5 Kipushi 2022 Feasibility Study

The Kipushi 2022 FS incorporates the Mineral Resource as reported in the March 2019 NI 43-101 Technical Report and these Mineral Resources remain unchanged and are valid and current for the purposes of this study.

Mining is planned to be a combination of transverse sublevel open stoping (SLOS) and pillar retreat mining methods. The Big Zinc area stopes are planned to be mined as SLOS to be extracted in a primary and secondary sequence, filled with cemented rock fill (CRF). The sill pillars are to be mined on retreat once the stopes below and above have been mined. The mine production is expected to be 0.8 Mtpa. Underground tonnes are anticipated to be mined, crushed in underground facilities, and hoisted to the surface via Shaft 5. The crushed material is expected to be pre-concentrated in a dense media separation (DMS) plant, followed by milling and flotation to produce saleable concentrate.

Life-of-mine (LOM) average annual planned zinc concentrate production is anticipated to be 437 ktpa, with a concentrate grade of 54.8% Zn. Total zinc ore production is anticipated to be 10.8 Mt at 31.85% Zn over a period of 14 years to produce 3,294 kt of zinc metal in concentrate.

Concentrate is planned to be transported from the Kipushi Mine by truck to the rail loading terminal at Ndola. The concentrate would then either be transported direct to the port on trucks, or trans loaded onto rail for transport to the export ocean port, from where it would be shipped by sea to customers.

The estimates of cash flows have been prepared on a real basis as at 1 January 2021 and a mid-year discounting is used to calculate Net Present Value (NPV). All monetary figures expressed in this report are US dollars ($) unless otherwise stated.

The economic analysis uses price assumptions of $2,646/t Zn. This price is based on a review of consensus price forecasts from a financial institutions and similar studies recently published.

1.6 Exploration and Drilling Programmes

Other than drilling, no other relevant exploration work has been carried out by KICO on the Kipushi Project.

1.6.1 Gécamines Drilling

Gécamines' drilling department (Mission de Sondages) historically carried out all drilling. Underground diamond drilling involved drill sections spaced 15 m apart along the Kipushi Fault Zone and Big Zinc and 12.5 m apart along the Série Récurrente, with each section consisting of a fan of between four and seven holes, the angle between holes being approximately 15°. Drilling was completed along the Fault Zone from Section 0 to Section 19 along a 285 m strike length including a 100–130 m strike length which also tested the Big Zinc. A total of 84 holes intersected the Big Zinc, of which 55 holes were surveyed downhole at a nominal 50 m spacing. Drill core from 49 of the 60 holes drilled from 1,272 mL which intersected the Big Zinc are stored under cover at the Kipushi Mine. Gécamines sampling tended to be based on individual samples representing mineable zones, with little attention paid to geology and mineralisation.

1.6.2 KICO Drilling

All work carried out during the two KICO underground drilling campaigns were performed according to documented standard operating procedures for the Kipushi Project. An original 25,400 m underground drilling programme was carried out by KICO between March 2014 and October 2015. A subsequent 9,700 m drilling campaign was carried out from May to October 2017. At the cut-off date of 24 April 2018, a total of 157 holes had been drilled (34,843 m), including 59 holes that intersected the Big Zinc, and 31 that intersected the Southern Zinc.

KICO's drilling was undertaken by Major Drilling SPRL from 1 March 2014 until the end of September 2014 when Titan Drilling Congo SARL took over diamond drilling operations for the remainder of the first drill programme, and the entire second drill programme. Drilling was completed using Boart Longyear LM75 and LM90 electro-hydraulic underground drill rigs.

Drilling was carried out on the same 15 m spaced sections used by Gécamines and comprised twin holes, infill holes and step-out resource definition holes.

Drilling was mostly NQ-TW (51 mm diameter) size with holes largely inclined downwards at various orientations to intersect specific targets within the Big Zinc, Fault Zone, Copper Nord Riche, and Série Récurrente. Along the section lines, the drillholes intersected mineralisation between 10–50 m apart within the Big Zinc and adjacent Fault Zone Mineral Resource area, and up to 100 m apart in the deeper parts of the Fault Zone outside of the Mineral Resource area.

Drilling has confirmed that zinc and copper mineralisation extend below the historical inferred resources to 1,825 m below surface with the deepest intersection recorded in hole KPU079. The Fault Zone is open at depth. Drilling from the second drill programme was successful in expanding the Southern Zinc and upgrading Inferred Mineral Resources to Indicated Mineral Resources for the Southern Zinc and Série Récurrente. Six of the holes drilled provided material for metallurgical testwork; one in the Nord Riche, two in the Fault Zone, one in the Série Récurrente and two in the Big Zinc.

1.7 Sample Preparation and Analysis

1.7.1 Gécamines Sample Preparation and Analysis

Historical sampling and assaying were carried out by Gécamines at the Kipushi laboratory. Sample analysis was carried out by a four-acid digest with AAS finish for Cu, Co, Zn, and Fe. The GBC Avanta AAS instrument originally used for the assays is still operational. Sulfur analysis was carried out by the 'classical' gravimetric method.

No data are available for quality assurance and quality control (QA/QC) protocols implemented for the Gécamines samples and therefore the Gécamines sample assays were considered to be less reliable than the KICO sample assays.

1.7.1.1 Resampling Programme

A comprehensive resampling programme was undertaken on historical Gécamines drill core from the Big Zinc and Fault Zone below 1,270 mL at the Kipushi Mine. The objectives of the exercise were to verify historical assay results and to quantify confidence in the historical assay database for its use in Mineral Resource estimation. In addition, KICO completed a number of twin holes on the Big Zinc between March 2014 and May 2015 with the objective of verifying historical Gécamines results. It was concluded that the results of the drill core resampling programme confirm that the assay values reported by Gécamines are reasonable and can be replicated within a reasonable level of error by international accredited laboratories under strict QA/QC control.

A total of 384 quarter core samples (NQ size core) were collected from historical Gécamines drill core and submitted to the KICO affiliated containerised sample preparation laboratory in Kolwezi for sample preparation. This facility and the sample preparation procedures were inspected for KICO by an independent consultant and found to be suitable for preparation of the Kipushi samples. A total of 457 samples including quality control (QC) samples were then submitted to the Bureau Veritas Minerals laboratory in Perth, Australia (BVM) for analysis. Density determinations on every tenth sample were carried out at BVM using the gas pycnometry method.

The final accepted Zn assays reported by BVM revealed an under-reporting by Gécamines for grades >25% Zn, and over-reporting at grades <20% Zn. Several outlier pairs were observed that are likely to result from mixed core or discrepancies in depth intervals, considering that the original drilling, sampling, and assay took place some 25 years ago. If the obvious outliers are excluded, the BVM results are, on average, 5.5% higher than the Gécamines results.

The observed discrepancies may be in part due to a difference in analytical approach, with the original assays having been carried out by Gécamines at the Kipushi laboratory by four-acid digest with AAS finish, for Cu, Co, Zn, and Fe rather than the Sodium Peroxide Fusion (SPF) method used by BVM.

Results for the other elements of interest are as follows:

- Several outlier pairs are observed in the Cu results that are likely to result from mixed core or discrepancies in depth intervals. Apart from the obvious outliers, a general correlation is observed between Gécamines and BVM that is considered acceptable, given the nuggety style of copper mineralisation.
- Disregarding the few outliers, BVM slightly under-reports Pb compared to Gécamines.
- S displays a similar pattern to Zn, with slight over-reporting at higher-grades and under-reporting at lower-grades by BVM compared to Gécamines.
- Gold was not routinely reported in historical assays but was reported as part of the resampling programme. Grades are typically low with a maximum of 0.21 ppm Au reported.

1.7.1.2 Density

As part of the historical data verification exercise, density determinations were carried out by gas pycnometry on every tenth sample at BVM resulting in a data set of 40 readings. In addition, density determinations using the water immersion (Archimedes) method were carried out on a representative piece of 15 cm drill core for each sample during the 2013 relogging campaign.

Gécamines used the following formula, derived mainly for the Fault Zone, to calculate density for use in historical tonnage estimates:

$$\text{Density} = 2.85 + 0.039 \times \text{Cu\%} + 0.0252 \times \text{Pb\%} + 0.0171 \times \text{Zn\%}$$

A comparison between density results (based on the Gécamines formula, laboratory gas pycnometry method, and the Archimedes method relative to zinc grade for the same samples showed that density, and hence tonnage, is understated by an average of 9% using the Gécamines calculated approach.

For the KICO drillholes, density was measured by KICO on whole lengths of half core samples using Archimedes principal of weight in air versus weight in water. Not all of the KICO samples were measured for density. A regression formula for density was derived from the KICO measurements to estimate the density of each sample based on its grade. This formula was applied to the Gécamines samples and those KICO samples that did not have density measurements.

1.7.2 KICO Sample Preparation and Analysis

All sample preparation, analyses and security measures were carried out under standard operating procedures set up by KICO for the Kipushi Project.

For drillholes KPU001 to KPU051, sample lengths were a nominal 1 m, but adjusted to smaller intervals to honour mineralisation styles and lithological contacts. From hole KPU051 onwards, the nominal sample length was adjusted to 2 m, with allowance for reduced sample lengths to honour mineralisation styles and lithological contacts. Following sample mark-up, the drill cores were cut longitudinally in half using a diamond saw. Half core samples were collected continuously through the identified mineralised zones.

Samples were dried at between 100°C and 105°C and crushed to a nominal 70% passing 2 mm, using either a TM Engineering manufactured Terminator jaw crusher or a Rocklabs Boyd jaw crusher. Subsamples (800–1,000 g) were collected by riffle splitting and milled to 90% passing 75 µm using Labtech Essa LM2 mills. Crushers and pulverisers were flushed with barren quartz material and cleaned with compressed air between each sample.

Grain size monitoring tests were conducted on samples labelled duplicates, which comprise about 5% of total samples, and the results recorded.

Subsamples collected for assaying and witness samples comprise the following:

- Three 40 g samples for DRC government agencies.
- A 140 g sample for assaying at BVM.
- A 40 g sample for portable XRF analyses.
- A 90 g sample for office archives.

The laboratory analytical approach and suite of elements for the underground drilling programme were informed by the results of:

- An 'orientation' exercise to confirm the analytical approach for a comprehensive resampling campaign on historical drill core and to characterise the major and trace element geochemistry of the Big Zinc deposit.
- Resampling of selected Gécamines drillholes which intersected the Fault Zone and Big Zinc.

The orientation samples were submitted to both BVM and Intertek Genalysis in Perth, Australia for analysis by SPF and ICP finish, high-grade and standard four acid digest with ICP finish, and gold by fire assay with AAS finish.

BVM was selected as the primary laboratory for the underground drilling programme, and representative pulverised subsamples from the underground drilling submitted for the following elements and assay methods, based on the results of the orientation sampling and resampling programmes:

- Zn, Cu, and S assays by SPF with ICP-OES finish.
- Pb, Ag, As, Cd, Co, Ge, Re, Ni, Mo, V, and U assays by peroxide fusion with ICP-MS finish.
- Ag and Hg by Aqua Regia digest with ICP-MS finish.
- Au, Pt, and Pd by 10 g (due to inherent high Sulfur content of the samples) lead collection fire assay with ICP-OES finish.

For silver, Aqua Regia assays were used below approximately 50 ppm and SPF assays were used above approximately 50 ppm.

A comprehensive chain of custody and QA/QC programme was maintained by KICO throughout the underground drilling campaign comprising drillholes KPU001 to KPU156. The QA/QC programme was established to monitor the quality of data for geological modelling and Mineral Resource estimation. All KICO data from the project are stored in an MS Access database. QA/QC data were exported from the MS Access database into software applications for creating monitoring charts and comparison charts.

The results of the QA/QC programme on recent drilling demonstrate that the quality of the assay data for zinc, copper, and lead is acceptable for supporting the estimation of Mineral Resources. Higher value data for silver, germanium, and gold are useable for resource estimation with some limitations.

1.8 Geology and Mineralisation

Kipushi is located within the Central African Copperbelt which constitutes a metallogenic province that hosts numerous world-class copper-cobalt deposits both in the DRC and Zambia. It is contained in the Katangan basin, an intracratonic rift that records onset of growth at ~840 Mpa and inversion at ~535 Mpa (Selley et al., 2018). The succession is divided into three regionally mappable groups, which from oldest to youngest are named the Roan, Nguba, and Kundelungu Groups.

The Kipushi Project is located within Nguba Group rocks on the northern limb of the regional west–north-west trending Kipushi Anticline which straddles the border between Zambia and the DRC. The mineral deposits at Kipushi are an example of carbonate-hosted copper–zinc–lead mineralisation hosted in pipe-like fault breccia zones, as well as tabular zones.

Mineralisation is focused at the intersection of the Kakontwe and Katete Formations of the Nguba Group with a north–north-east striking 70° west dipping discontinuity known as the Kipushi Fault, which terminates the northern limb of the anticline. The Kipushi Fault has been interpreted by KICO as a syn-sedimentary reef-edge environment, with possible reactivation during the Lufilian Orogeny. Mineralisation occurs in several distinct settings known as the Fault Zone (copper, zinc, and mixed copper–zinc mineralisation both as massive sulfides and as veins), the Copper Nord Riche (mainly copper but also mixed copper–zinc mineralisation, both massive and vein-style), the Série Récurrente (disseminated to veinlet-style copper mineralisation), the Big Zinc (massive zinc with local copper mineralisation), and the Southern Zinc (polymetallic zone with massive zinc and copper mineralisation).

Copper-dominant mineralisation in the form of chalcopyrite, bornite, and tennantite is characteristically associated with dolomitic shales both within the Fault Zone and extending eastwards along, and parallel to bedding planes within the Katete Formation. Zinc-dominant mineralisation in the Kakontwe Formation occurs as massive, irregular, discordant pipe-like bodies replacing the dolomite host, which exhibit a steep southerly plunge from the Fault Zone and Série Récurrente contacts where they begin, to their terminations at depth within the Kakontwe Formation.

1.9 Metallurgical Testwork Summary

Metallurgical testwork programme were completed on drill core samples of known Kipushi mineralisation between 2013–2018 for the various project redevelopment study phases. These investigations were focused on metallurgical characterisation and flow sheet development for the processing of material from the Big Zinc orebody.

In 2016, a Pre-economic Assessment (PEA) study was executed which focussed primarily on the processing of the Big Zinc utilising gravity processes. It was proposed to process the ore in coarse and fine fractions, through DMS and spiral facilities. The DMS facility was split into a coarse and fine DMS circuit, processing the –20 mm +6 mm and –6 mm +1 mm fraction respectively. The –1 mm fraction reported to the gravity spiral circuit.

Two sets of metallurgical testing were conducted in support of the PEA study. These included:

- Kipushi 2013 Scoping Study:

  A testwork campaign in 2013 was executed on a 60 kg sample derived from a composite of quarter-core pieces. The average grade of the sample was 38% Zn, 12% Fe, 0.78% Pb and 0.4% Cu. The sample was predominantly sphalerite and pyrite, with minor amounts of galena and chalcopyrite. Scoping work that was conducted on this sample was promising and prompted the receipt of an additional fresh sample for testing.

- Resource Development Phase – 2015:

  In 2015, a fresh metallurgical sample was obtained for further testing as part of resource model development. The sample comprised of six drill cores intercepting the Big Zinc and represented the major mineralisation types. The target head grades for the composite was 37% Zn. The major economic minerals within the composite were sphalerite (67%), galena (2%) and chalcopyrite (1%). Predominant non-sulfide gangue minerals included dolomite and quartz, with pyrite representing the major sulfide gangue phase.

The outcome of this study was that the composite could be satisfactorily upgraded at a coarse size fraction, utilising DMS, where stage recovery in the region of 99% Zn was attained at concentrate grades of 55% Zn, at a cut point of 3.1 t/m^3. It was noted that this result was consistently obtained at all three size fractions tested. The processing of the fines fraction was less successful, resulting in a stage recovery of 58% of Zn in feed at a final grade of approximately 56% Zn. As such, the metallurgical performance of the sample with the coarsest top size was marginally better than other top sizes, purely because the ratio of coarse to fine material in the feed was less than other samples tested.

Overall metallurgical performance for the final flow sheet was an overall Zn recovery of 95.4%, with a final concentrate grade of 55.5% Zn. This formed the basis for the PEA process flow sheet.

1.9.1 Kipushi 2017 PFS Metallurgical Testwork

Early into the 2017 Pre-feasibility Study (PFS), it was identified that the proposed circuit had some limitations in terms of performance, stemming from the gangue mineralogy. It was established that:

- The gravity circuits were highly effective at rejecting non-sulfide gangue, such as dolomite, as the density differential between the ore and non-sulfide gangue was large. As such, recovery to the final DMS concentrate for the DMS was excellent, and
- Gravity separation processes concentrate all sulfides including pyrite (a major gangue phase in the ore), chalcopyrite and galena. None of these minerals is rejected in either the spirals or the DMS.

As a result, the concentrate grades obtained by both DMS and spirals would be driven by the proportion of pyrite in the feed, and through variability testwork and METSIM simulations conducted at the time, it was demonstrated that the required saleable concentrate grade would not be attainable by this method.

The process flow sheet evolved from the proposed gravity circuit to use hybrid DMS/flotation circuits in order to provide a suitable mechanism for pyrite rejection. Testwork to support economic evaluation of these options was conducted on a new master composite. This composite was made up of four drill cores intercepting the Big Zinc, which were selected to represent all styles of mineralisation present in the orebody. The assayed intervals were composited in such a manner that a master composite with a head grade of 32–33% Zn could be produced. This grade range represented the average run-of-mine (ROM) Zn grade, based on the PFS mine plan.

The PFS programme confirmed that with PFS levels of pyrite in plant feed the proposed coarse / fine gravity separation-based circuit would not be able to produce concentrates above the 53% Zn threshold. This is because gravity separation processes were limited to removing non-sulfide gangue from the ore.

Flotation was essential to attain the necessary upgrades. The development of a promising bulk sulfide flotation regime occurred very late in the PFS test programme but was not able to be optimised. As such, the PFS study was based on a more comprehensively tested sequential flotation process flow sheet. It was recommended that bulk flotation be evaluated further in the Feasibility Study (FS).

1.9.2 Kipushi 2022 FS Metallurgical Testwork

Two dedicated metallurgical drill cores, KPU101 and KPU104 provided 18 drill core sections which were selected based on continuity of ore type. The 18 samples were combined, based on their dominant ore types, to form 10 ore type composites. An FS flow sheet development composite of average grade was prepared from the 10 ore type composites. In addition, a hand-picked dolomite waste sample was prepared.

As ore types will always be presented to the plant in mixtures, nine plant feed variability composites were prepared from the 10 ore type composites. Samples were selected to represent various combinations of Fe, Pb, Cu and Zn falling within the spread of compositions in the monthly average LOM schedule.

Nine comminution samples selected from the same drill cores were subjected to Uniaxial compressive strength (UCS) and Bond Crusher Work Index (CWi) testing. Based on these results, four samples were selected for SAG milling comminution (SMC) testing. Bond work indices (Ai, BRWi and BBWi) were conducted on all nine samples.

Compressive Strength of all samples are classed as hard or very hard. Measured values ranged from 152 MPa (KPU104MSM) in ore to 282 MPa for the dolomite rich sample (KPU104SDO2).

Bond Crusher Work Index (CWi) values are highly variable. All but one of the samples were classified within the very soft range (<10 kWh/t), with the exception of KPUSDO1 which was classified in the soft range (10–14 kWh/t range).

Abrasion index (Ai) results ranged from slightly abrasive (0.1–0.4) to non-abrasive (less than 0.1). Only four samples had Ai values greater than 0.1.

Bond Rod Work Index (BRWi) results were soft to medium hardness, ranging from 7.9 kWh/t to 14 kWh/t. Dolomite (SDO) and Mixed Sulfide Minerals (MSM) ore types are potentially harder than the MBS ore types, but this judgement is based on a limited sample set. The 75th percentile value for the BRWI was 12.7 kWh/t with an average value of 10.7 kWh/t.

The Bond Ball Work Index (BBWi) results were soft with respect to ball milling. The BBWi ranged from 6.6–9.6 kWh/t. The 75th percentile value for the Bond ball work index was 8.73 kWh/t with an average value of 8.02 kWh/t.

The range of Axb values (by SMC testing) was broad, with the most competent sample reporting a medium competence Axb value of 51 (KPU104SDO2) and the least competent sample reporting an Axb value of 142.1 (KPU104MBS2).

The FS master composite was split into two samples. The first was used for ROM flotation testwork and the second for sequential DMS-flotation testwork. Flotation scope included baseline testing of both sequential and bulk sulfide flotation, grind optimisation and some reagent optimisation.

The DMS-flotation sample was crushed to –12 mm then the –12 +1 fraction was separated using heavy liquids to identify an optimal cut point of 3.1 t/m^3. A ferrosilicon DMS pilot run was performed targeting the optimal cut-point of 3.1 SG. The DMS concentrate was crushed to –1 mm and was blended with the –1 mm fraction for flotation testing. DMS floats were prepared and sent away for cemented rock fill tests.

Once the optimal flow sheet was established, bulk flotation testwork was conducted to generate samples for thickening, filtration, and environmental studies.

The outcome of the heavy liquid separation (HLS) testing on the –12 mm +1 mm fraction confirmed the outcomes of testwork in the PFS phase:

- Mass rejection of approximately 30% could be achieved at a cut point of 3.1 t/m^3.
- At this cut point, Zn recovery to the sinks fraction was as high as 98.94%.
- Changing the top size of the DMS feed from –20 mm to –12 mm had a negligible impact on overall Zn recovery.

Metallurgical performance, in terms of Grade/Recovery relationships for all sequential flotation testwork conducted on the ROM ore, none of the non-cyanide based sequential reagent systems could produce better metallurgical performance than the baseline test. It was noted that increasing the depressant dosage produced superior performance.

It was also noted that the bulk sulfide testwork could not produce a high-grade final Zn concentrate, and this could not be remedied through the use of the proposed pyrite depressants. The Zn concentrate grade could not be improved by making use of an additional cleaner stage. The bulk flotation regime could also not improve upon the Zn recovery obtained by the baseline cyanide based sequential flotation circuit.

Process optimisation testing found that none of the conditions tested could improve upon metallurgical performance achieved by the baseline bulk flotation test.

To test the effect of site water on metallurgical performance, samples of water from the Kipushi site were furnished for testwork. It was noted that the use of site water impacted significantly on overall recovery, with final concentrate of 54% Zn attainable at recoveries as low as 86.5%. For the project, the decision was therefore made to utilise potable water for make-up. The tailings return water is also not returned to the process circuit.

A bulk sample of flotation concentrate was supplied to Outotec and to Tenova Delkor to conduct testwork required for sizing of the concentrate thickener, as well as most appropriate flocculant and optimal dose thereof.

Estimated concentrate quality in terms of chemical composition, and associated penalties for off-specification products was derived from the chemical analysis of bulk concentrates produced during the testwork campaign from the FS master composite. The results showed that under average ROM conditions, the concentrate meets the requirements in terms of deleterious elements. However, the Fe and Mg grades of this concentrate are within close range of the prescribed limits for these elements, and as such, penalties could be incurred should the feed grades increase substantially from the average. In addition to this, it should be noted that, because a bulk sulfide concentrate is produced, the concentrate grade will be sensitive to Cu and Pb feed grades, as well as pyrite content thus close control will be required to ensure that the targeted concentrate quality is achieved.

Upgrade testing was conducted for each of the nine variability samples using DMS followed by bulk flotation to reject pyrite. The DMS results showed that recovery of Zinc, and all other sulfides, was almost complete as expected from the FS composite results. Initial bulk flotation results showed poor upgrade for very low feed samples, with none of the samples achieving the 53% grade threshold. Some of the lower grade samples did not achieve 50% Zn, and this was typically linked to the zinc head grade. For example, samples 5 and 8, 19% and 15% Zn in feed, achieved only 36% Zn and 47% Zn respectively in concentrate. Repeat testing was ordered to determine if the flotation test method was at fault rather than the selectivity of flotation.

Repeat testing on sample 2 increased concentrate grade from 52% to almost 56%. Similar testing on sample 4 raised concentrate grade from 50–54%. The difference between the tests was to operate at lower pulp level and lower air rate compared to the initial variability tests. These changes promoted better froth drainage and avoided pulp overflow with the froth. Although not exhaustively tested, the repeat work has given confidence that the minimum grade of 53% Zn will be achievable on a shipment basis and that the design grade of 54.8% is supportable given the high average feed grade of the ore and provided the flotation equipment and configuration is designed with due care.

The flotation variability data was sufficient to define a predictive mass pull vs recovery relationship. The model was extended to produce a head grade vs recovery relationship for a target concentrate grade of 54.8% Zn. Both relationships have been used in project evaluation.

The flotation model was then utilised to produce a head grade vs recovery relationship for a target concentrate grade of 54.79% Zn. The overall recovery model was built by producing mass and metal balances for a range of Zn head grades to obtain the target concentrate grade. The produced overall Zn recovery as a function of head grade was fitted to obtain the overall head grade/ recovery relationship for the project.

1.10 Mineral Resource Estimates

The Mineral Resource estimate has an effective date of 14 June 2018 and represents an update to the previous Mineral Resource estimate (effective date of 23 January 2016) as a result of additional diamond drilling completed by KICO from May–November 2017. The recent drilling was focussed on infill and extension in the Southern Zinc / Fault Zone and Série Récurrente areas, and extensions to the Big Zinc and the Série Récurrente footwall massive sulfide zone. The infill and extension drilling programme provided a further 41 mineralised core intersections to those completed by KICO from March 2014–November 2017 that were used in the 23 January 2016 estimate.

In addition to the KICO drillholes, Gécamines drilled numerous diamond drillholes during the operational period of the mine. A number of the Gécamines holes were examined and re-sampled and a database was compiled from the historical data by MSA. A programme of twin and infill drilling demonstrated that the Gécamines data were overall unbiased compared to the KICO data and where the quality of the data was considered acceptable it was incorporated into the Mineral Resource estimate.

In total, 106 Gécamines holes and 134 KICO holes were used for the grade estimate. The cut-off date for data included in this estimate is 26 April 2018, there being no additional drilling data collected since then.

The Mineral Resource was estimated using The Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Best Practice Guidelines and is reported in accordance with the 2014 CIM Definition Standards, which have been incorporated by reference into National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101).

The Mineral Resources were categorised either as zinc-rich resources or copper-rich resources, depending on the most abundant metal. For the zinc-rich zones (Big Zinc and Southern Zinc) the Mineral Resource is reported at a base case cut-off grade of 7.0% Zn and the copper-rich zones (Fault Zone, Fault Zone Splay and Série Récurrente) at a base case cut-off grade of 1.5% Cu.

The Mineral Resource is classified into the Measured, Indicated and Inferred categories as shown in Table 1.1 for the predominantly zinc-rich bodies and in Table 1.2 for the predominantly copper-rich bodies.

Given the considerable revenue which will be obtained from the additional metals in each zone, MSA considers that mineralisation at these cut-off grades will satisfy reasonable prospects for eventual economic extraction. It should be noted that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability and the economic parameters used to assess the potential for economic extraction is not an attempt to estimate Mineral Reserves.

Table 1.1 Kipushi Zinc-Rich Mineral Resource at 7% Zn Cut-off Grade, 14 June 2018

Zone	Category	Tonnes (Mt)	Zn (%)	Cu (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
Big Zinc	Measured	3.65	39.87	0.65	0.35	18	18	56
	Indicated	7.25	34.36	0.62	1.29	19	12	53
	Inferred	0.98	35.32	1.18	0.09	8	15	62
Southern Zinc	Indicated	0.88	24.52	2.97	1.95	75	6	188
	Inferred	0.16	24.37	1.64	1.20	38	6	61
Total	Measured	3.65	39.87	0.65	0.35	18	18	56
	Indicated	8.13	33.30	0.87	1.36	25	11	68
	Measured and Indicated	11.78	35.34	0.80	1.05	23	13	64
	Inferred	1.14	33.77	1.24	0.24	12	14	62
Contained Metal Quantities								
Zone	Category	Tonnes (Mt)	Zn (Mlb)	Cu (Mlb)	Pb (Mlb)	Ag (Moz)	Co (Mlb)	Ge (Moz)
Big Zinc	Measured	3.65	3,210.6	52.3	27.8	2.06	0.14	6.60
	Indicated	7.25	5,489.0	98.7	206.6	4.48	0.19	12.43
	Inferred	0.98	764.0	25.5	1.9	0.26	0.03	1.96
Southern Zinc	Indicated	0.88	476.5	57.6	37.8	2.11	0.01	5.34
	Inferred	0.16	86.7	5.8	4.3	0.20	0.00	0.32
Total	Measured	3.65	3,210.6	52.3	27.8	2.06	0.14	6.60
	Indicated	8.13	5,965.5	156.4	244.4	6.59	0.20	17.77
	Measured and Indicated	11.78	9,176.0	208.6	272.2	8.65	0.34	24.36
	Inferred	1.14	850.7	31.3	6.2	0.46	0.04	2.28

1. All tabulated data has been rounded and as a result minor computational errors may occur.
2. Mineral Resources that are not Mineral Reserves, have no demonstrated economic viability.
3. The Mineral Resource is reported as the total in-situ Mineral Resource and on a 100% project basis, exclusive of Mineral Reserves. Ivanhoe holds an indirect 68% interest in the Project.
4. Metal quantities are reported in multiples of Troy Ounces or Avoirdupois Pounds.
5. The cut-off grade calculation was based on the following assumptions: zinc price of $1.00/lb, mining cost of $50/t, processing cost of $10/t, G&A and holding cost of $10/t, transport of 55% Zn concentrate at $210/t, 90% zinc recovery and 85% payable zinc.

Table 1.2 Kipushi Copper-Rich Mineral Resource at 1.5% Cu cut-off grade, 14 June 2018

Zone	Category	Tonnes (Mt)	Zn (%)	Cu (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
Fault Zone	Measured	0.14	1.52	2.74	0.04	16	77	21
	Indicated	1.22	3.32	4.11	0.09	21	96	30
	Inferred	0.20	2.58	3.11	0.07	18	43	23
Série Récurrente	Indicated	0.93	2.43	4.14	0.02	23	50	4
	Inferred	0.03	0.06	1.81	0.00	8	52	0.3
Fault Zone Splay	Inferred	0.21	19.84	4.91	0.01	21	107	93
Total	Measured	0.14	1.52	2.74	0.04	16	77	21
	Indicated	2.15	2.94	4.12	0.06	22	76	19
	Measured and Indicated	2.29	2.85	4.03	0.06	21	76	19
	Inferred	0.44	10.77	3.89	0.04	19	75	55
Contained Metal Quantities								
Zone	Category	Tonnes (Mt)	Zn (Mlb)	Cu (Mlb)	Pb (Mlb)	Ag (Moz)	Co (Mlb)	Ge (Moz)
Fault Zone	Measured	0.14	4.7	8.5	0.1	0.07	0.02	0.09
	Indicated	1.22	89.7	110.8	2.5	0.82	0.26	1.19
	Inferred	0.20	11.1	13.4	0.3	0.12	0.02	0.14
Série Récurrente	Indicated	0.93	49.8	84.6	0.5	0.69	0.10	0.12
	Inferred	0.03	0.04	1.3	0.0	0.01	0.00	0.00
Fault Zone Splay	Inferred	0.21	93.7	23.2	0.1	0.14	0.05	0.64
Total	Measured	0.14	4.7	8.5	0.1	0.07	0.02	0.09
	Indicated	2.15	139.4	195.4	3.0	1.51	0.36	1.31
	Measured and Indicated	2.29	144.2	204.0	3.1	1.58	0.39	1.40
	Inferred	0.44	104.9	37.9	0.4	0.27	0.07	0.78

1. All tabulated data has been rounded and as a result minor computational errors may occur.
2. Mineral Resources which are not Mineral Reserves, have no demonstrated economic viability.
3. The Mineral Resource is reported as the total in-situ Mineral Resource and on a 100% project basis, exclusive of Mineral Reserves. Ivanhoe holds an indirect 68% interest in the Project.
4. Metal quantities are reported in multiples of Troy Ounces or Avoirdupois Pounds.
5. The cut-off grade calculation was based on the following assumptions: copper price of $3.00/lb, mining cost of $50/t, processing cost of $10/t, G&A and holding cost of $10/t, 90% copper recovery and 96% payable copper.

The Measured and Indicated Mineral Resource for the zinc-rich bodies have been tabulated using a number of cut-off grades as shown in Table 1.3, and the Inferred Mineral Resource in Table 1.4.

Table 1.3 Kipushi Zinc-Rich Bodies Measured and Indicated Mineral Resource Grade Tonnage Table, 14 June 2018

Cut-off (Zn%)	Tonnes (Mt)	Zn (%)	Contained Zn (Mlb)	Cu (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
5	11.91	35.01	9,193.7	0.81	1.04	23	13	64
7	11.78	35.34	9,176.0	0.80	1.05	23	13	64
10	11.51	35.96	9,125.4	0.78	1.06	23	13	65
12	11.26	36.52	9,063.5	0.76	1.06	23	13	65
15	10.83	37.42	8,937.0	0.73	1.06	23	13	65

1. All tabulated data has been rounded and as a result minor computational errors may occur.
2. Mineral Resources are not Mineral Reserves, have no demonstrated economic viability.
3. The Mineral Resource is reported as the total in-situ Mineral Resource and on a 100% project basis, exclusive of Mineral Reserves. Ivanhoe holds an indirect 68% interest in the Project.
4. Metal quantities are reported in multiples of Troy Ounces or Avoirdupois Pounds.

Table 1.4 Kipushi Zinc-Rich Bodies Inferred Mineral Resource Grade Tonnage Table, 14 June 2018

Cut-off (Zn%)	Tonnes (Mt)	Zn (%)	Contained Zn (Mlb)	Cu (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
5	1.14	33.77	850.7	1.24	0.24	12	14	62
7	1.14	33.77	850.7	1.24	0.24	12	14	62
10	1.14	33.78	850.6	1.24	0.24	12	14	62
12	1.14	33.91	849.0	1.24	0.24	12	14	61
15	1.11	34.29	842.7	1.21	0.23	12	14	61

1. All tabulated data has been rounded and as a result minor computational errors may occur.
2. Mineral Resources are not Mineral Reserves, have no demonstrated economic viability.
3. The Mineral Resource is reported as the total in-situ Mineral Resource and on a 100% project basis, exclusive of Mineral Reserves. Ivanhoe holds an indirect 68% interest in the Project.
4. Metal quantities are reported in multiples of Troy Ounces or Avoirdupois Pounds.

The Measured and Indicated Mineral Resource for the copper-rich bodies have been tabulated using a number of cut-off grades as shown in Table 1.5, and the Inferred Mineral Resource in Table 1.6.

Table 1.5 Kipushi Copper-Rich Bodies Measured and Indicated Mineral Resource Grade Tonnage Table, 14 June 2018

Cut-off (Cu %)	Tonnes (Mt)	Cu (%)	Contained Cu (Mlb)	Zn (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
1.0	3.72	2.96	242.6	2.10	0.04	17	58	14
1.5	2.29	4.03	204.0	2.85	0.06	21	76	19
2.0	1.55	5.16	175.7	3.59	0.08	26	93	23
2.5	1.20	5.99	158.9	4.08	0.09	30	107	26
3.0	1.00	6.65	146.7	4.43	0.09	33	118	26

1. All tabulated data has been rounded and as a result minor computational errors may occur.
2. Mineral Resources are not Mineral Reserves, have no demonstrated economic viability.
3. The Mineral Resource is reported as the total in-situ Mineral Resource and on a 100% project basis, exclusive of Mineral Reserves. Ivanhoe holds an indirect 68% interest in the Project.
4. Metal quantities are reported in multiples of Troy Ounces or Avoirdupois Pounds.

Table 1.6 Kipushi Copper-Rich Bodies Inferred Mineral Resource Grade Tonnage Table, 14 June 2018

Cut-off (Cu %)	Tonnes (Mt)	Cu (%)	Contained Cu (Mlb)	Zn (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
1.0	0.55	3.39	40.8	11.90	0.03	17	66	64
1.5	0.44	3.89	37.9	10.77	0.04	19	75	55
2.0	0.35	4.49	34.3	12.21	0.03	20	84	61
2.5	0.29	4.93	31.5	12.14	0.03	21	92	58
3.0	0.24	5.38	28.6	11.18	0.02	22	100	53

1. All tabulated data has been rounded and as a result minor computational errors may occur.
2. Mineral Resources are not Mineral Reserves, have no demonstrated economic viability.
3. The Mineral Resource is reported as the total in-situ Mineral Resource and on a 100% project basis, exclusive of Mineral Reserves. Ivanhoe holds an indirect 68% interest in the Project.
4. Metal quantities are reported in multiples of Troy Ounces or Avoirdupois Pounds.

Mineral Resource estimates were completed below the 1,150 mL on the Big Zinc, Southern Zinc, Fault Zone and Série Récurrente, extensive mining having taken place in the levels above. Below 1,150 mL, some mining has taken place, which has been depleted from the model for reporting of the Mineral Resource. The maximum depth of the Mineral Resource of 1810 mL is dictated by the location of the diamond drilling data. The Mineral Resource occurs close to the DRC-Zambia Border and the Mineral Resource has been constrained to the area considered to be within the DRC.

The Mineral Resource estimate has been completed by Mr. J.C. Witley (BSc Hons, MSc (Eng)) who is a geologist with 30 years' experience in base and precious metals exploration and mining as well as Mineral Resource evaluation and reporting. He is a Principal Mineral Resource Consultant for The MSA Group (an independent consulting company), is registered with the South African Council for Natural Scientific Professions (SACNASP) and is a Fellow of the Geological Society of South Africa (GSSA). Mr. Witley has the appropriate relevant qualifications and experience to be considered a 'Qualified Person' for the style and type of mineralisation and activity being undertaken as defined in National Instrument 43-101 Standards of Disclosure of Mineral Projects.

1.11 Mineral Reserves

The Kipushi 2019 Mineral Reserve is defined by a feasibility level study. The Kipushi 2022 FS Mineral Reserve has been estimated by Qualified Person Bernard Peters, Technical Director – Mining, OreWin, using the 2014 CIM Definition Standards. The Mineral Reserve is based on the 14 June 2018 Mineral Resource. The effective date of the Mineral Reserve statement is 14 February 2022. Table 1.7 shows the total Proven and Probable Mineral Reserve of Kipushi.

Table 1.7 Kipushi Proven and Probable Reserve – Tonnage and Grades

Category	Tonnage (Mt)	Zn (%)	Contained Zn (kt)
Proven	3.33	37.4	1,246
Probable	7.48	29.4	2,199
Total	10.82	31.9	3,445

1. The effective date of the Mineral Reserves is 14 February 2022.
2. Net Smelter Return (NSR) is used to define the Mineral Reserve cut-offs, therefore cut-off is denominated in \$/t. By definition, the cut-off is the point at which the costs are equal to the NSR. An elevated cut-off grade of \$135/t NSR was used to define the mining shapes. The marginal cut-off grade has been calculated to be \$50/t NSR.
3. The Kipushi 2022 FS Mineral Reserve is based on a zinc price of \$1.10/lb Zn and a treatment charge of \$170/t concentrate, while the economic analysis to demonstrate the Kipushi 2022 FS Mineral Reserve has used a zinc price of \$1.20/lb Zn and a treatment charge of \$190/t concentrate.
4. Only Measured Mineral Resources were used to report Proven Mineral Reserves and only Indicated Mineral Resources were used to report Probable Mineral Reserves.
5. Mineral Reserves reported above were not additive to the Mineral Resources and are quoted on a 100% project basis.
6. The Mineral Reserve is based on the 14 June 2018 Mineral Resource.
7. Totals may not match due to rounding.
8. The Proven and Probable Reserve estimate has been reported to conform with the CIM Standards on Mineral Resources (CIM, 2005) of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM).

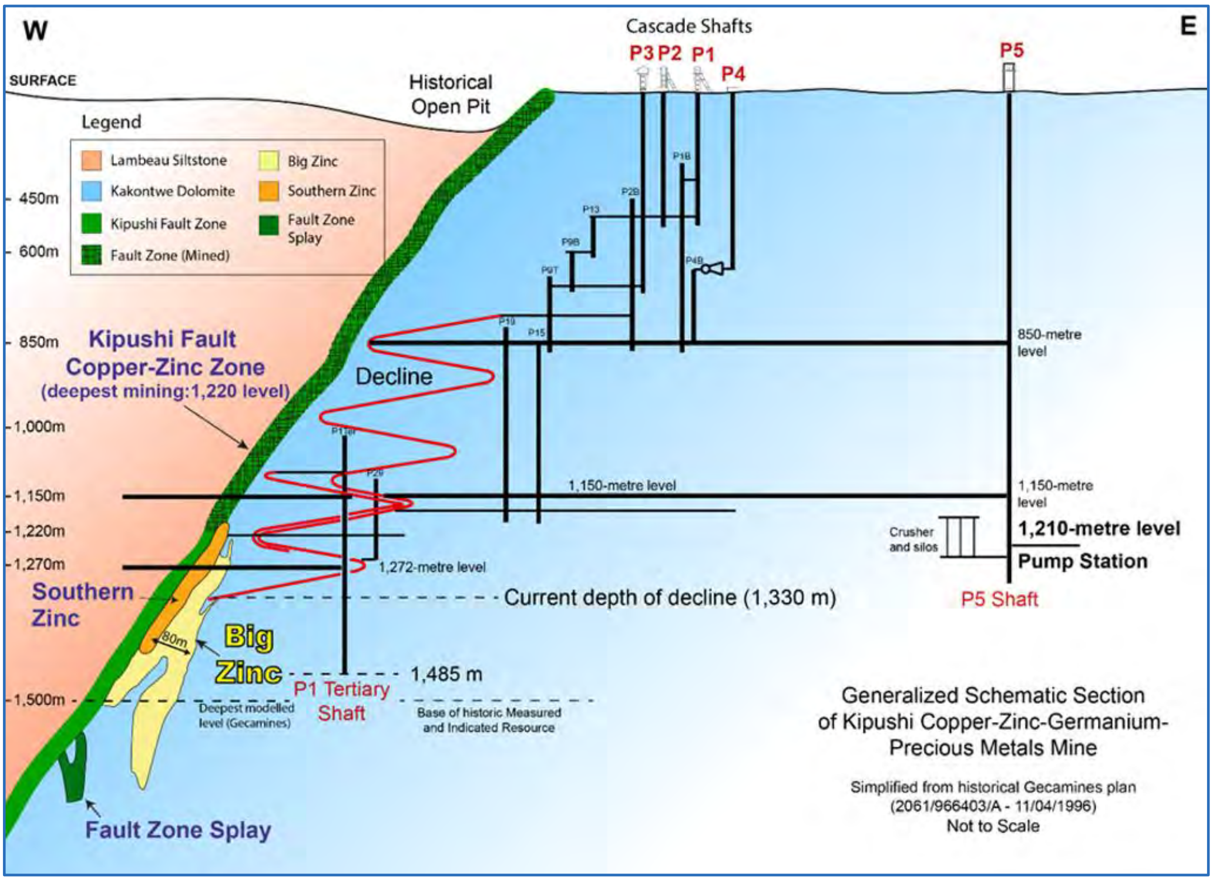
1.12 Mining

Historical mining at Kipushi was carried out from the surface to approximately 1,220 m mL and occurred in three contiguous zones: The North and South zones of the Fault Zone, and the Série Récurrente in the footwall of the fault that is approximately east–west striking and steeply north dipping.

KICO has a significant amount of underground infrastructure at the Kipushi Project, including a series of vertical mine shafts, with associated headframes, to various depths, as well as underground mine excavations. A schematic layout of the existing development is shown in Figure 1.1.

The newest shaft, Shaft 5 (labelled as P5 in Figure 1.1 below) is 8 m in diameter and 1,240 m deep. Shaft 5 is planned to be the main production shaft. It has a maximum hoisting capacity of 1.8 Mtpa and provides the primary access to the lower levels of the mine, including the Big Zinc, through the 1,150 mL haulage level. Shaft 5 is approximately 1.5 km from the main mining area. All three headframe mounted hoists have been upgraded and commissioned. A series of crosscuts and ventilation infrastructure are still in working condition. The underground infrastructure also includes a series of pumps to manage the influx of water into the mine.

Figure 1.1 Schematic Section of Kipushi Mine



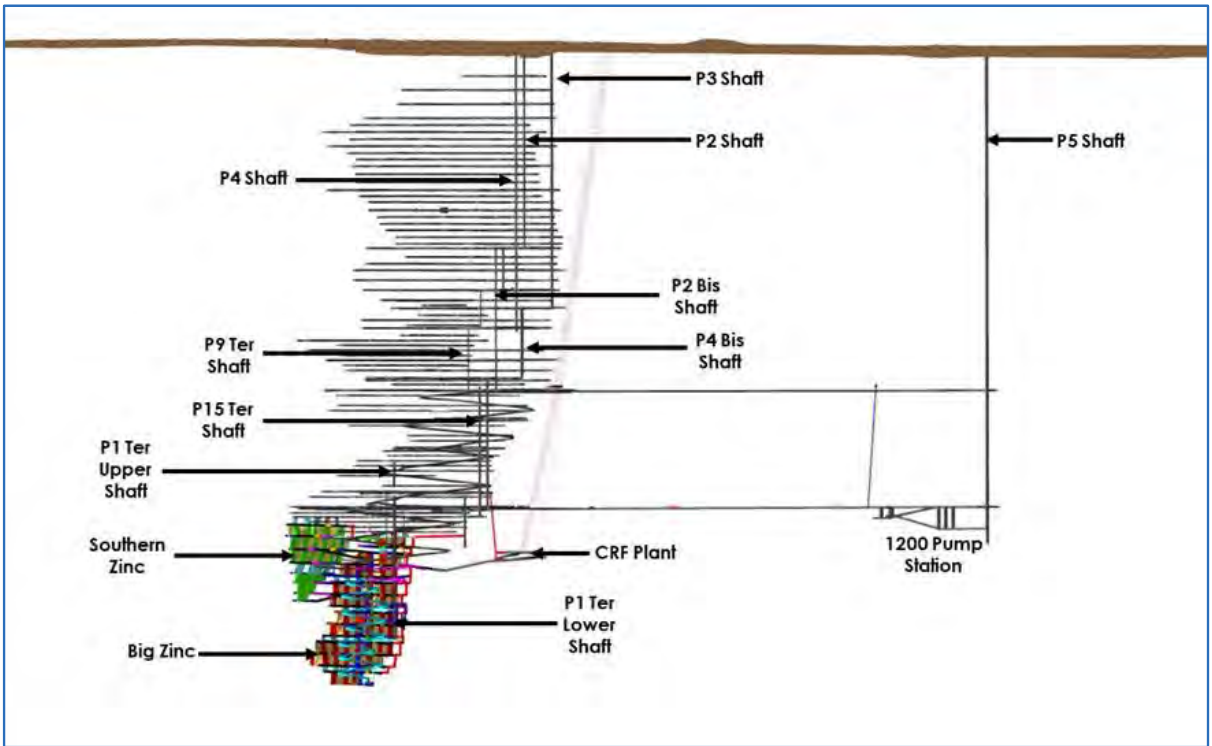
Ivanhoe, 2022

Mining zones included in the current Kipushi Mine plans occur at depths ranging from approximately 1,207 mL and 1,590 mL with 0 mL being the surface. Access to the mine will be via existing multiple vertical shafts and internal decline. Mining will be performed using highly productive mechanised methods and CRF backfill will be utilised to fill open stopes. Depending on required composition and available material, excess waste rock and, DMS tailings will be used in the CRF mix as required.

The CRF is an engineered material, which will allow both the large wall exposures when mining the secondary stopes and undercuts when mining the tertiaries. The fill will be batched underground in an underground fill plant and transported and placed in the stopes using underground dump trucks and loaders. Fill materials (aggregates and cement slurry) will be delivered down fill and aggregate passes from the surface to the underground fill plant. Where possible, the aggregates on the surface will be sourced from waste generated underground from the mining of waste development drives and from reject material from the DMS step in the process plant. Mining is planned to be a combination of transverse SLOS and pillar retreat mining methods. The Big Zinc area stopes are planned to be mined as SLOS to be extracted in a primary and secondary sequence, filled with CRF. The sill pillars are to be mined on retreat once the stopes below and above have been mined.

Material generated underground will be trucked to the base of the P5 shaft, crushed and hoisted to surface. Personnel and equipment access are also via the P5 shaft. The Big Zinc is expected to be accessed via the existing decline and without significant new development. The decline is planned to be developed from the existing level at approximately 1,330 mL to the bottom stoping level at 1,590 mL. The zinc stoping is expected to be carried out between 1,207 mL and 1,590 mL, and the uppermost stoping level on the Big Zinc is planned to be the 1,245 mL. As the existing decline is already below the first planned stoping level, there is potential to develop the first zinc stopes early in the mining schedule which could achieve a rapid ramp up of mine production. The main access levels are planned to be at 60 m vertical intervals with sublevels at 30 m intervals. The sill pillar height is planned to be 15 m. Stopes are planned to be mined 60 m along strike and then filled with CRF. Remote capable loaders are expected to be used for loading the broken rock beyond the stope brow. The existing and planned development and stoping is shown in Figure 1.2.

Figure 1.2 Planned Kipushi 2022 FS and Existing Development



OreWin, 2022

1.13 Processing Plant

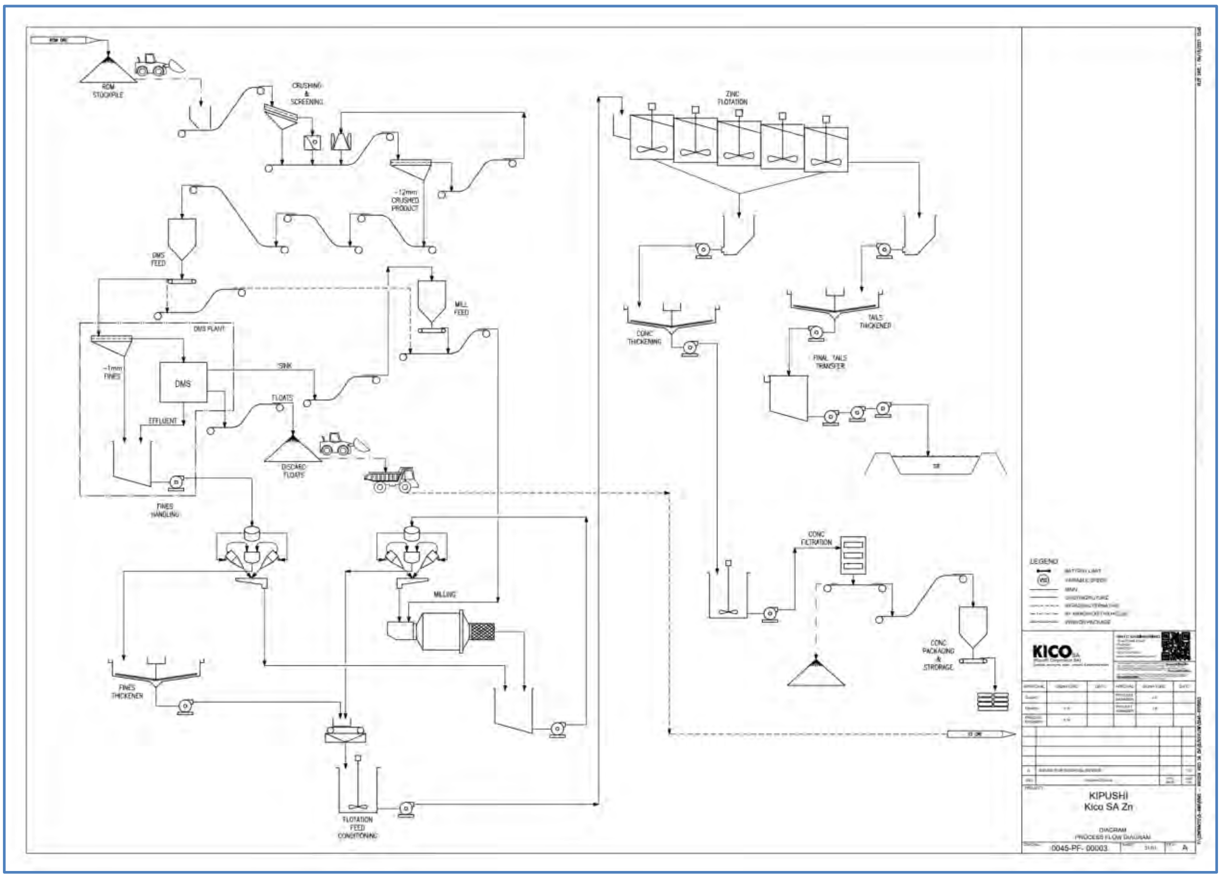
The concentrator plant is designed to process a nominal 800 ktpa of run-of-mine (ROM) ore, from the high-grade Big Zinc orebody of the Kipushi underground Zinc-Copper Mine in the Central African Copperbelt in the DRC. The metallurgical tests on representative samples showed that the Kipushi mineralisation is amenable to a process involving pre-concentration by dense media separation (DMS) and bulk flotation to produce suitable saleable zinc sulfide concentrate.

Ore and waste from the Big Zinc is crushed underground to a product size of 100% passing 280 mm and hoisted to surface using Shaft 5. Both crushed ore and development waste will be intermittently (and separately) hoisted to surface and transferred to the stockpiles area. Ore is withdrawn from the stockpile by means of front-end loaders (FEL) to feed the process plant, whilst reclaimed waste material is loaded into trucks for disposal.

The proposed overall process flow circuit includes ROM crushing and screening, pre-concentration by DMS, ball mill grinding circuit, sulfide flotation, final tailings and concentrate handling facilities, air utilities, reagent, and water services. The crushing plant product particle top size is set at 12 mm, whilst the milling circuit final grind size is at 80% passing 106 µm.

The proposed flow sheet is illustrated in Figure 1.3, whilst the processing route employed is summarised below.

Figure 1.3 Kipushi – Concentrator Plant Block Flow Diagram



METC, 2022

The crushing and screening circuit receives ROM ore onto the ore stockpile, via a series of overland transfer conveyors, as primary crusher product, from the Kipushi underground mine.

Ore is withdrawn from the ore stockpile using a FEL to feed a two-stage crushing plant at a nominal solids feed rate of 140 t/h. The crusher circuit design has been set up to minimise the production of fines (–1 mm). To this end, an open circuit secondary jaw crusher is used in conjunction with a closed-circuit tertiary cone crusher.

Screened crushed ore product, at –12 mm size fraction, is transferred and discharged into a DMS feed bin from where the DMS plant is fed.

Crushed ore from the DMS feed bin is withdrawn and screened to produce a –12 + 1 mm size fraction product to feed the DMS. Screen undersize as –1 mm fines, together with the DMS effluent slurry, is pumped to a dewatering cyclone cluster at the fines handling circuit. The fines handling dewatering cyclone overflow gravitate to the fines thickener feed box whilst the cyclone underflow gravitates to the mill discharge sump. The fines thickener feed is thickened to 55% w/w solids and transferred to the flotation conditioning tank via a trash linear screen. Design allows for a flexibility to pump thickener underflow to the mill discharge sump.

The DMS feed, at an average grade of 31.9% w/w Zn, is upgraded to about 47% w/w Zn in concentrate at an average mass pull of approximately 71% w/w to sinks product. The DMS product is transferred via a conveyor to the mill feed bin. The DMS floats discard is conveyed to the DMS discard stockpile, from where material for CRF is collected.

The milling circuit is designed as a closed-circuit variable speed ball mill with a classification cyclone cluster. The milling circuit comprises a single 1,1 MW ball mill. The milling circuit is designed to achieve a P_{80} of 106 µm. The cyclone overflow gravitates to the flotation feed conditioning tank via a trash linear screen.

The flotation circuit comprises a single bank of 5 off 20 m^3 identical flotation cells sized to achieve a design residence time of 25 minutes, dedicated concentrate, and tails pump transfer systems, as well as the blower air supply system and the reagent dosing systems.

The flotation circuit is operated at pH of 11.5 to depress pyrite from floating with the concentrate. The reagent regime includes a xanthate collector (SIPX), copper sulfate to activate the sphalerite and a frother to assist with stable froth formation.

The Zn flotation concentrate is thickened and filtered ahead of bagging in a semi-automated bagging facility. Concentrate thickener underflow is filtered using a vertical tower filter press to produce a concentrate filter cake with 9% moisture content. The concentrate bags are stored in dedicated storage bays with facility to load onto trucks. Each bag is sampled and tagged ahead of dispatch.

Flotation tailings stream is thickened and pumped to the tailings storage facility (TSF). Both tailings thickener overflow and concentrate thickener overflow water solutions are transferred to the process water circuit. Two main fresh water sources for the operation of the process plant exist, namely:

- Mine dewatering water, pumped through P5 shaft mining facilities.
- Potable water, sourced from the existing well field water supply line.

The underground mine dewatering water quality is not suited for use in concrete making for the CRF facility, fire water or gland service water. Metallurgical testwork also indicated that underground water negatively impacted metallurgical performance, and therefore could not be utilised as process water make-up. The potable water quality from the existing borehole well field is considered adequate for process top-up water, gland service water, and reagent make-up water requirements in the process plant.

1.14 Environmental and Social Impact Assessment

In accordance with the terms of Article 463 of the Decree No. 18/024 of 8 June 2018 modifying and completing the Decree No. 038/2003 of 26 March 2003 on Mining Regulations, the holder of a mining right is required to revise its approved Environmental and Social Impact Assessment (ESIA) and Environmental and Social Management Plan (ESMP) every five years, or when changes in mining or quarrying activities justify a change in the Environmental and Social Impact Assessment. KICO, as the promoter of this project, therefore requested the services of an Environmental Design Office approved in DRC, Congo Environment and Mining Consulting (CEMIC SARL), to conduct an updated ESIA, as well as to verify and/or to evaluate the implementation of the proposed mitigation and rehabilitation plans in the ESMP.

CEMIC SARL have been appointed to update the approved Kipushi Project ESIA/ESMP, in accordance with the prescribed requirements set out in Annex VIII of the 2018 Mining Regulations (Decree No. 018/024 of 8 June 2018). The directive relating to its development prescribed in Annex VIII of the Mining Regulations includes eight main points mentioned as follows:

1. Awareness of the directive on the environmental and social impact assessment during the preparation of the latter.
2. Presentation of the project (project details).
3. Analysis of the environmental system affected by the project.
4. Analysis of the environmental impacts of operations.
5. Rehabilitation and mitigation measures programme.
6. Detailed budget and financial plan for the rehabilitation and mitigation programme as well as the financial guarantee for the environmental rehabilitation.
7. Consultation of the population during the preparation of community Environmental and Social Impact Assessment and of the sustainable development plan.
8. The certification of compliance.

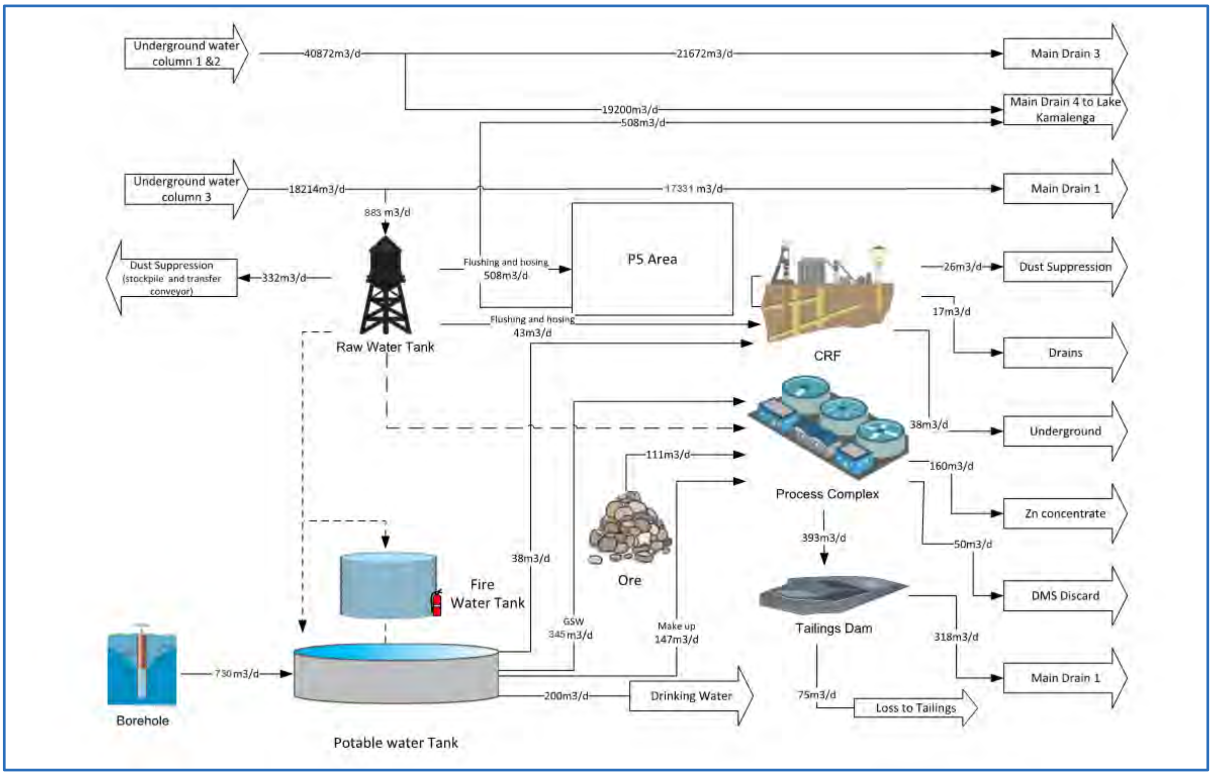
1.15 Water Management

As part of the FS, Golder Associates Africa updated the Surface Water Impact Assessment for the Kipushi Project. This included a baseline and impact assessment. As part of the impact assessment, potential surface water impacts were identified on, and external to the project area. The identified impacts were assessed and mitigation strategies formulated, taking into consideration the TSF design, dewatering and future proposed surface water infrastructure.

Metallurgical testwork conducted indicates that the raw water recovered from mine dewatering is not appropriate for use in the flotation circuit. Therefore, potable water is utilised as process water make-up. Provision is made, however, for make up using raw water if required. Flotation tailings will be deposited in a new TSF located south of the process plant as shown in Figure 1.4. In the proposed scheme, the return from the TSF is first neutralised and blended with the excess underground water before discharging to the Kipushi River via the north cut-off channel. A neutralisation plant has been included in the FS as the geochemical analysis undertaken on the basis of available data indicated possible acidity of the TSF return water that, even after blending with underground water, falls outside DRC prescribed discharge limits.

A system of clean water channels has been designed to cut-off the clean run-off upstream of the TSF. The clean water is returned to the environment.

Figure 1.4 Water Management Block Flow Diagram



Golder, 2019

1.16 Tailings Management and Disposal

The TSF has been designed to contain the tailings stream based on a ROM of 181 ktpa and a LOM of 13.8 years. The expected total production of dry tailings are:

- 2.6 Mt of Zinc tailings, relating to.
- 1.3 Mm³ of Zinc Tailings.

Several sites were provisionally identified as potential sites for location of the TSF as shown in Figure 1.5 A ranking matrix identified Site 4 as the most optimal location for the TSF. However, during detailed engineering and execution, the KICO team will also be considering Site 1.

Figure 1.5 FS Tailings Dam Locations – Site 4 Selected for the Study

Epoch, 2019

The TSF will be constructed in two phases with the containment walls being developed in a downstream construction method. Phasing the construction allows for early tailings storage volume without requiring the entire developed facility, while construction of a successive phase may continue, if the timeline requires it, and defers the CAPEX throughout the LOM.

The TSF will comprise the following:

- A phased compacted earth-fill full containment wall with material sourced from the TSF basin or approved borrow areas.
- Single liner system comprising of:
  - Base preparation of the in-situ material which will be ripped to depth of 300 mm and re-compacted to 98% of the Standard Proctor density.
  - An A6 Bidim geotextile (or similar specified) secured in a liner anchor trench.
  - A 1,500 micron HDPE (High-Density Polyethylene) geomembrane across the basin and side slopes secured in a liner anchor trench.

- A phased curtain drain constructed in the containment walls of Phase 1 and 2, comprising a 160 mm slotted HDPE Drainex pipe within a coarse stone matrix overlain by an intermediate stone, overlain by a filter sand and protected by an A6 geotextile (or similar specified) on the downstream face with the upstream face exposed to the compacted impoundment wall material.
- A curtain drain outlet comprising of a 160 mm non-slotted HDPE Drainex pipe within a coarse stone matrix wrapped in an A6 Bidim geotextile with a tie-in to a pre-cast concrete collection manhole.

Supernatant water is to be decanted from the facility pool and either returned to the process plant or released into the environment by means of floating barge and pump system designed, supplied, installed, and operated by others.

1.17 Infrastructure

The property hosts surface mining and processing infrastructure, offices, workshops, stores, and connection to the national power grid. The property also hosts historical infrastructure, such as a mineral processing/beneficiation plant, that will not be further utilised. Most of the surface infrastructure was owned by Gécamines and was either ceded or leased to KICO. The overall proposed site layout is shown in Figure 1.6.

Key aspects of the Project infrastructure are:

- Electricity is supplied by the state power company of the DRC, Société Nationale d'Electricité (SNEL), using two transmission lines from Lubumbashi. There are pylons in place for a third line. The lines will be refurbished and re-stringed with aluminium conductors to minimise copper theft incidents.
- 14 MW of back-up power will be provided on site (new diesel gensets).
- The refurbishment of the diesel tank farm.
- Communications infrastructure required to support an operating mine.
- Leased and refurbished accommodation in Kipushi for owner's team personnel.
- A new overland conveyor for transporting ore and waste from Shaft 5, to the new plant/ore stockpile and temporary waste storage area, respectively.
- A ROM ore stockpile and a temporary waste stockpile area.
- A new processing plant and supporting surface infrastructure that incorporates the following unit operations:
  - Crushing and screening
  - DMS to remove dolomitic wastes for backfill
  - Milling
  - Flotation
  - Concentrate bagging facility
- A new tailings dam with an overhead line supplying power to the facility.

- A new backfill plant and supporting surface infrastructure that incorporates the following unit operations:
  - Crushing and screening
  - Cement silos/storage
  - Cement slurry plant
- A new border post and infrastructure to facilitate exports of zinc concentrate from the Kipushi border along the road to Solwezi in Zambia.
- A combination of old (refurbished) and new facilities including:
  - General office, technical buildings, and structures.
  - Mine services buildings (change rooms, mess, kitchen, laundry).
  - Workshops, stores, and construction laydown areas.
  - General electrical buildings.
  - Security and emergency services buildings.

Figure 1.6 Overall Proposed Site Layout

Ivanhoe, 2022

1.18 Concentrate Transport and Logistics

Given the saturated roads and border crossings, a sustainable logistics solution for Kipushi is critical for the viability of the mine project and continued stability of existing freight flows in and out of the Copperbelt.

From Kipushi to an ocean seaport there are various established road corridors within the Southern Africa Development Community (SADC) region. All of these routes are supported and promoted by the SADC Secretariat as part of their regional trade development commitment, and harmonisation of Customs border procedures is an ongoing process within the region.

The PFS developed a detailed supply chain model to calculate the transport costs and included an assessment of the various prevalent ground conditions to finally deliver a recommendation of the most viable transport option based on a holistic, quantitative, and qualitative evaluation. Typical ground conditions include, regulatory restrictions, border crossing limitations, availability of rolling stock, operational efficiencies, and port handling capabilities, including shipping. The PFS identified rail as being the preferred option for transporting zinc concentrate from the mine to a port of loading.

For the FS, a new study was commissioned to develop alternate supply chain options, bypassing the rail network, and to evaluate the associated costs/benefits according to the very same criteria as in the PFS study.

After comparison of the results with the outcomes of the PFS, the FS study identified road transport and multimodal (road to rail) supply chain, as the preferred options for transporting zinc concentrate from the mine to a port of loading. Concentrate trucks from Kipushi would utilise an existing road via Solwezi in Zambia to Ndola, which is undergoing refurbishment. The road originates at the mine and links Zambian T5 network near Kansanshi mine. This option would require the Kipushi border to be upgraded to facilitate mineral exports (currently only facilitates light vehicles). At Ndola the concentrate is then either transported direct to port via trucks or trans-loaded onto rail, utilising the North–South Rail Corridor from Ndola to Durban. The North–South Rail Corridor to Durban via Zimbabwe is fully operational and has significant excess capacity.

The refurbishment of the road from Kipushi is ongoing with recent upgrades including the graveling and widening the road. Trucks can then transport concentrate via Kansanshi to Ndola, where the concentrate will either be transported direct to the port on trucks, or trans-loaded onto rail for transport to the export ocean port, as shown in Figure 1.7.

As a base case for concentrate transport, the FS has assumed truck haulage direct to port as break bulk concentrate out of either Durban, Walvis Bay or Dar es Salaam to China (Shanghai).

Figure 1.7 Site to Ndola Concentrate Trucking Route

Ivanhoe, 2022

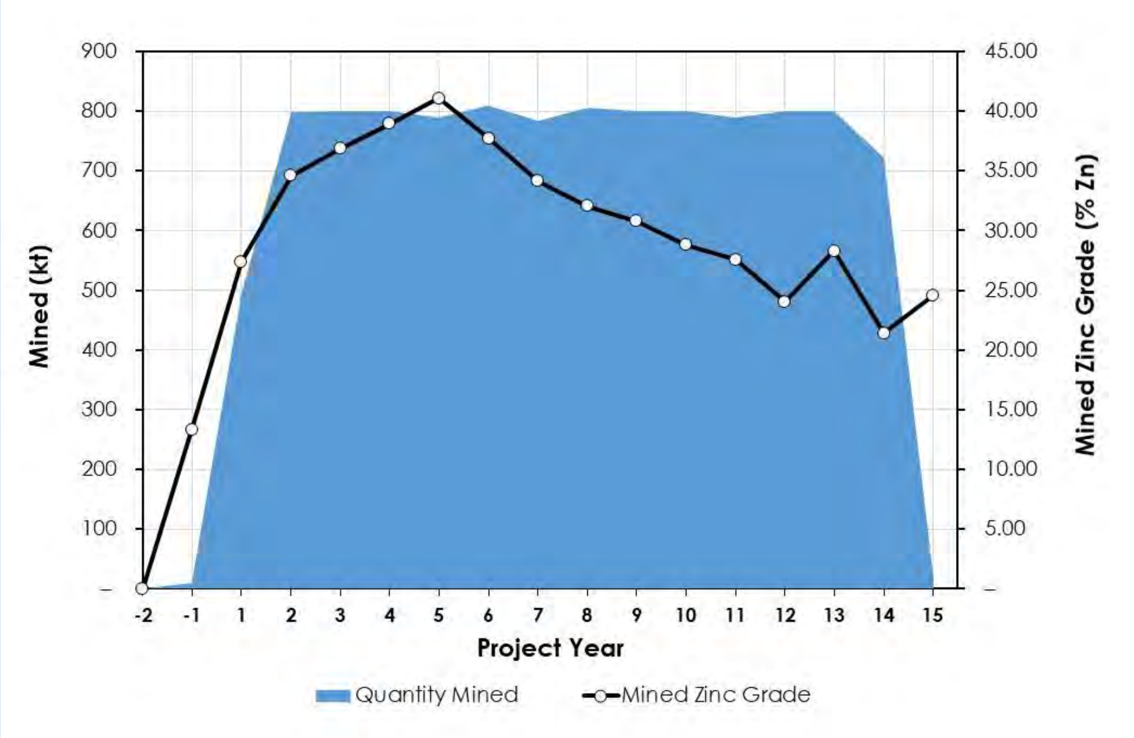
1.19 Production

Future proposed mine production has been scheduled to optimise the mine output and meet the plant capacity. The mining production forecasts are shown in Table 1.8. Mine, process and concentrate production are shown in Figure 1.8 to Figure 1.10.

Table 1.8 Mining Production Statistics

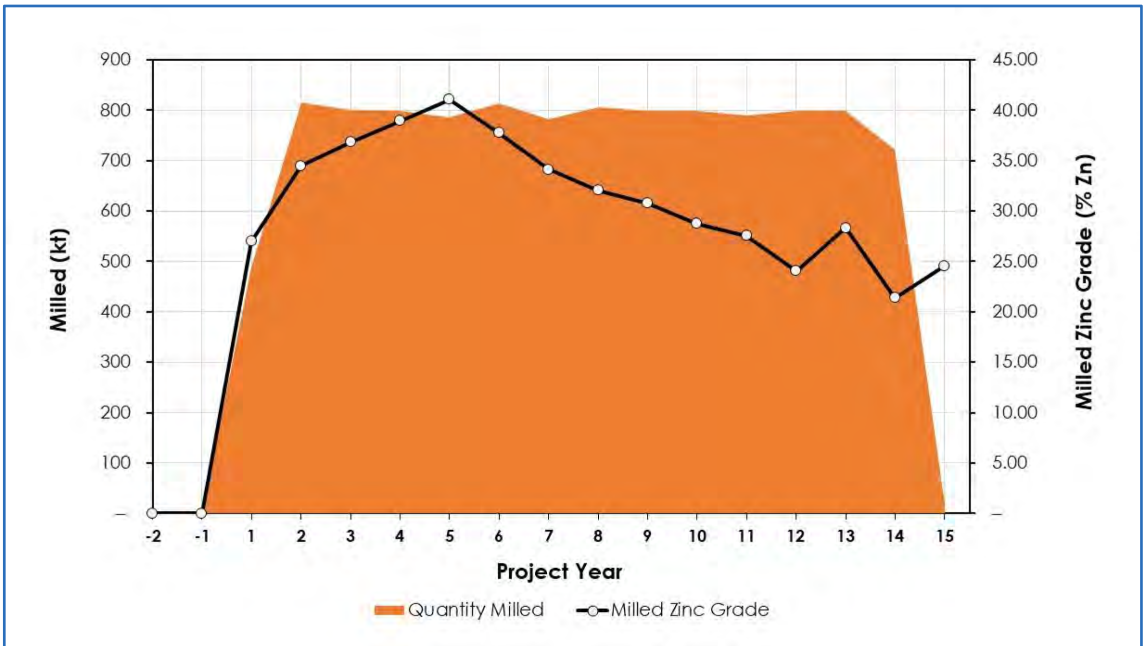
Item	Unit	Total LOM	5-Year Average	LOM Annual Average
Zinc Ore Processed				
Quantity Zinc Ore Treated	kt	10,814	792	787
Zinc Feed grade	%	31.85	36.43	31.85
Zinc Concentrate Recovery	%	95.63	95.87	95.63
Zinc Concentrate Produced	kt (dry)	6,013	508	437
Zinc Concentrate Grade	%	54.79	54.79	54.79
Metal Produced				
Zinc	kt	3,294	278	240

Figure 1.8 Mined Production



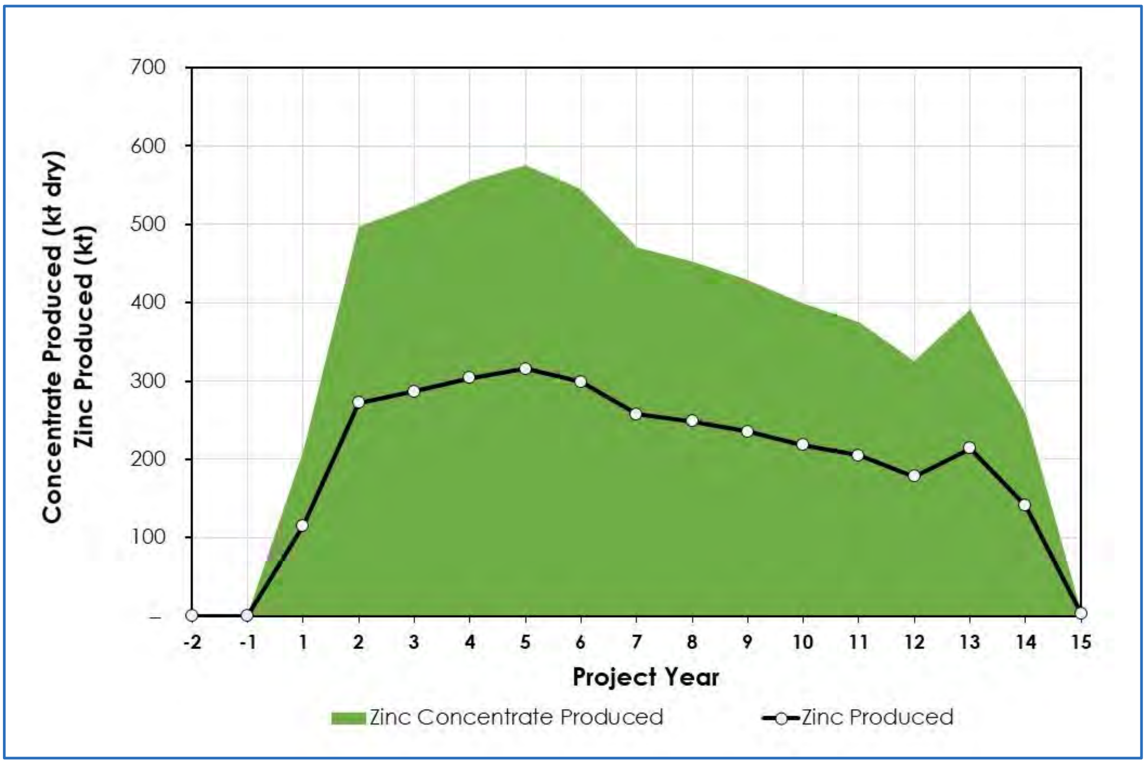
OreWin, 2022

Figure 1.9 Process Production



OreWin, 2022

Figure 1.10 Concentrate and Metal Production



OreWin, 2022

1.20 Economic Analysis

The estimates of cash flows have been prepared on a real basis as at 1 January 2022 and a mid-year discounting is used to calculate NPV.

The projected financial results include:

- After-tax NPV at an 8% real discount rate is $941M.
- After-tax internal rate of return (IRR) is 41%.
- After-tax project payback period is 2.33 years.

The projected financial results for undiscounted and discounted cash flows, at a range of discount rates, IRR and payback are shown in Table 1.9. The key economic assumptions for the discounted cash flow analyses are shown in Table 1.10. The results of NPV8% sensitivity analysis to a range of zinc prices and discount rates is shown in Table 1.11. The results of NPV8% and IRR sensitivity analysis to a range of zinc prices and zinc concentrate treatment charge is shown in Table 1.12.

A chart of the cumulative cash flow is shown in Figure 1.11.

Table 1.9 Financial Results

	Discount Rate	Before-tax	After-tax
Net Present Value ($M)	Undiscounted	2,452	1,946
	5.0%	1,536	1,228
	8.0%	1,175	941
	10.0%	986	790
	12.0%	829	663
	15.0%	640	510
	18.0%	495	391
	20.0%	415	326
Internal Rate of Return	–	44%	41%
Project Payback Period	–	2.3 years	2.3 years

Table 1.10 Metal Prices and Terms

Parameter	Unit	Financial Analysis Assumption
Zinc Price	$/lb	1.20
Zinc Treatment Charge	$/t concentrate	190

Table 1.11 After-Tax NPV8% Sensitivity to Zinc Price and Discount Rates

Discount Rate (%)	Zinc ($/lb)								
	0.80	1.00	1.10	1.20	1.30	1.40	1.60	1.80	2.00
Undiscounted	-100	1,052	1,478	1,856	2,247	2,642	3,809	3,022	3,830
5%	-160	600	901	1,166	1,434	1,701	2,478	2,954	3,757
8%	-184	422	671	890	1,109	1,326	1,949	2,885	3,678
10%	-196	329	550	745	937	1,128	1,671	2,817	3,586
12%	-205	253	450	624	794	963	1,439	2,748	3,494
15%	-216	162	328	477	621	763	1,158	2,679	3,402
18%	-223	92	235	362	486	606	939	2,562	3,249
20%	-226	55	184	300	412	521	819	3,022	3,830

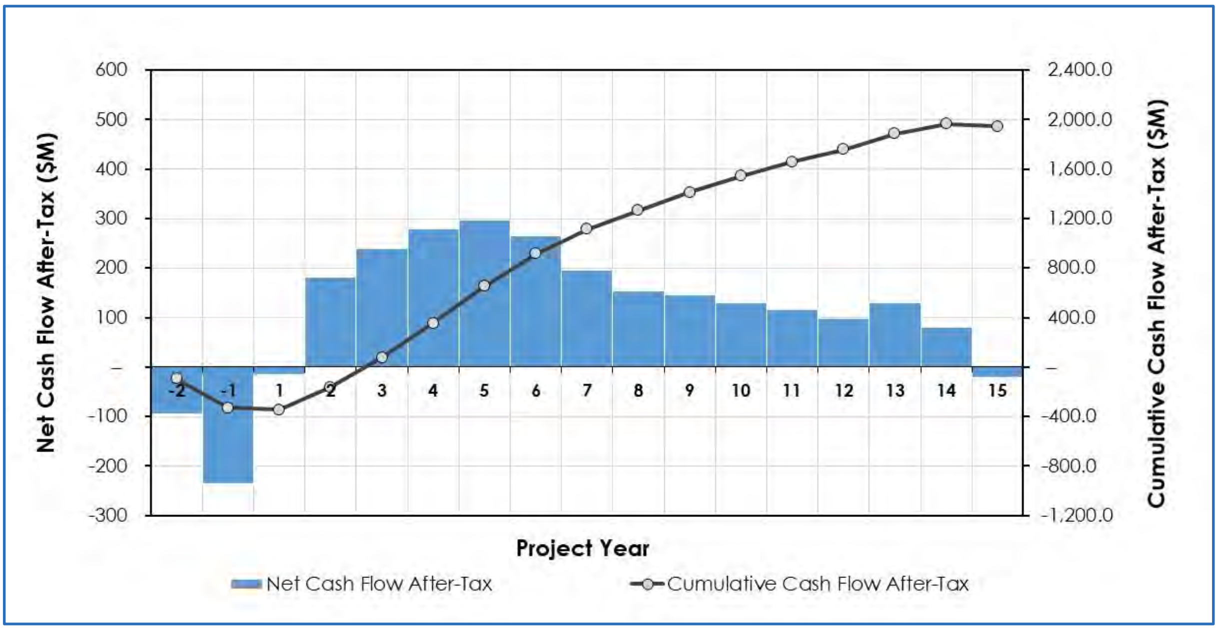
Table shows NPV8% $M.

Table 1.12 After-Tax NPV8% and IRR Sensitivity to Zinc Price and Zinc Treatment Charge

Zinc Treatment Charge ($/t)	Zinc Price ($/lb)								
	0.80	1.00	1.10	1.20	1.30	1.40	1.60	1.80	2.00
100	90	651	871	1,090	1,306	1,524	2,151	3,022	3,830
	12%	31%	39%	45%	52%	58%	72%	88%	101%
130	-4	584	805	1,023	1,241	1,458	2,081	2,954	3,757
	8%	29%	37%	43%	50%	56%	70%	86%	100%
160	-92	505	738	957	1,175	1,392	2,015	2,885	3,678
	3%	26%	34%	41%	48%	54%	68%	85%	98%
190	-184	422	671	890	1,109	1,326	1,949	2,817	3,586
	-6%	24%	32%	39%	46%	52%	67%	83%	97%
220	-277	329	603	824	1,042	1,260	1,882	2,748	3,494
	N/A	21%	30%	37%	44%	50%	65%	82%	95%
250	-368	236	529	757	976	1,194	1,816	2,679	3,402
	N/A	18%	27%	35%	42%	48%	63%	80%	94%
300	-482	81	387	645	865	1,084	1,706	2,562	3,249
	N/A	12%	23%	31%	38%	45%	60%	77%	91%

Table shows NPV8% $M and IRR.

Figure 1.11 Cumulative Cash Flow



OreWin, 2022

The total capital cost estimates for the Kipushi Project are shown in Table 1.13.

The estimated revenues and operating costs are presented in Table 1.14 along with the estimated net sales revenue value attributable to each key period of operation. The estimated cash costs are presented in Table 1.15.

Table 1.13 Total Project Capital Costs

Item	Pre-production ($M)	Production ($M)	Total ($M)
Mining			
Underground Mining	154	118	272
Capitalised Mining Operating Costs	17	–	17
Subtotal	171	118	289
Process and Infrastructure			
Process and Infrastructure	82	36	117
Road Rehabilitation	1	5	6
Tailings Storage Facility	7	9	15
Capitalised Processing	5	–	5
Subtotal	94	49	143
Closure			
Closure	–	22	22
TSF Closure and Rehabilitation	–	3	3
Subtotal	–	25	25
Indirects and Others			
EPCM	13	–	13
Total Owners Costs	47	–	47
Capitalised General and Administration	9	–	9
Customs Duties	2	3	6
VAT	21	-37	-16
Subtotal	89	-34	55
Capital Cost Before Contingency	57	159	516
Contingency	24	–	24
Capital Cost After Contingency	382	159	540

Table 1.14 Operating Costs and Revenues

Description	Total ($M)	5-Year Average ($/t Milled)	LOM Average ($/t Milled)
Revenue			
Gross Sales Revenue	7,408	790	685
Less Realisation Costs			
Transport Costs	1,757	187	162
Treatment and Refining Charges	1,142	122	106
Royalties	405	43	37
Total Realisation Costs	3,305	352	306
Net Sales Revenue	4,103	438	379
Less Site Operating Costs			
Total Mining	634	61	59
Processing Zn	248	23	23
General and Administration	201	20	19
Tailings Storage Facility	9	1	1
Customs Duties	19	2	2
Total	1,111	107	103
Operating Margin ($M)	2,992	331	277
Operating Margin (%)	40.4	42	40

Table 1.15 Cash Costs

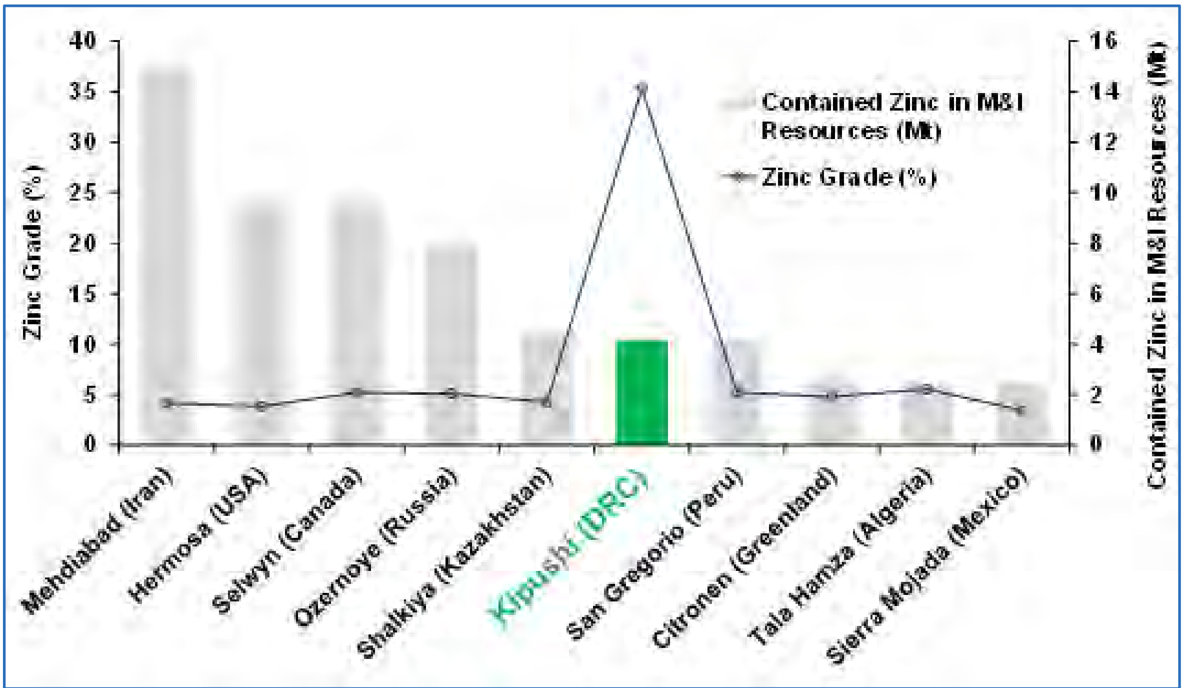
Description	5-Year Average Payable Zn ($/lb)	LOM Average Payable Zn ($/lb)
Mine Site Cash Cost	0.16	0.18
Transport Costs	0.28	0.28
Treatment and Refining Charges	0.19	0.19
C1 Cash Cost	0.63	0.65
Royalties	0.07	0.07
Total Cash Costs	0.70	0.72

1.20.1 Comparison to Other Projects

Using data for other zinc projects provided by Wood Mackenzie comparisons with the Kipushi 2022 FS were made for the following results: contained zinc in resources, production, capital intensity and C1 Cash Costs.

The Kipushi Project Mineral Resource Estimate, June 2018 includes Measured and Indicated Resources of 11.8 Mt at 35.34% Zn. This grade is more than twice as high as the Measured and Indicated Mineral Resources of the world's next-highest-grade zinc project, according to Wood Mackenzie, a leading, international industry research and consulting group (Figure 1.12).

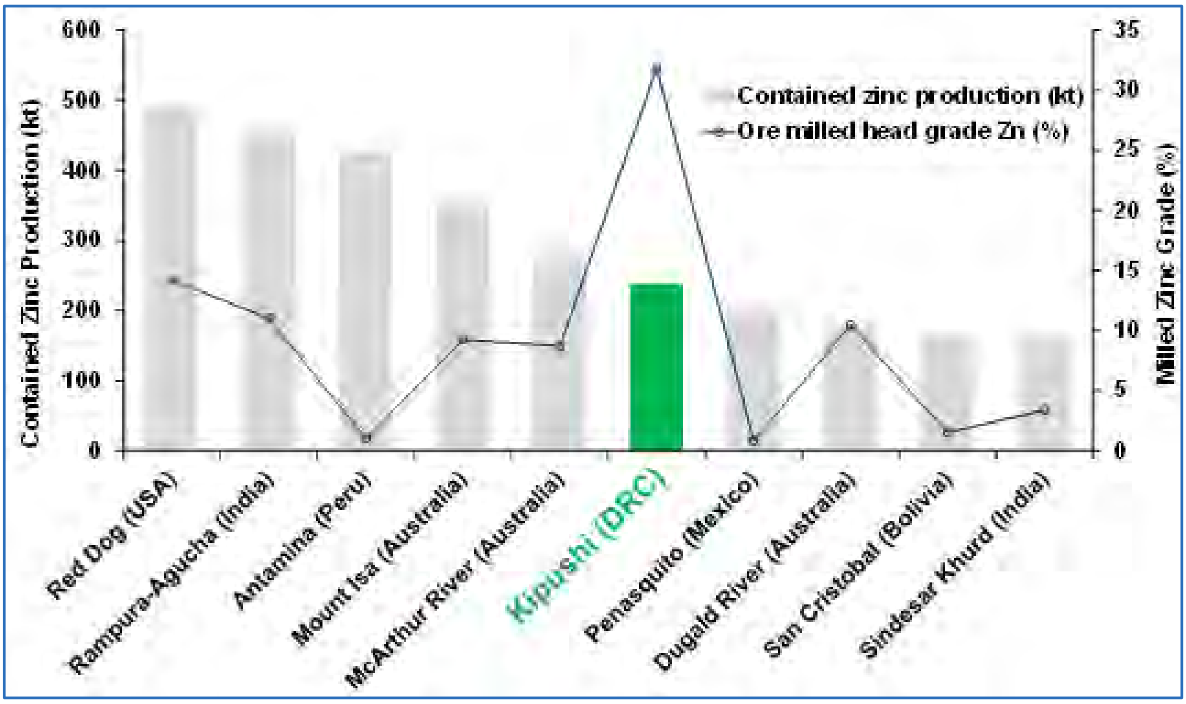
Figure 1.12 Top 20 Zinc Projects by Contained Zinc



Wood Mackenzie, 2022

LOM average planned zinc concentrate production of 437 ktpa, with a concentrate grade of 54.8% Zn, is expected to rank the Kipushi Project, once in production, among the world's major zinc mines (Figure 1.13). Based on research by Wood Mackenzie the world's major zinc mines defined as the world's 10 largest zinc mines ranked by production in 2020.

Figure 1.13 Major Zinc Mines Estimated 2020 Annual Zinc Production and Grade

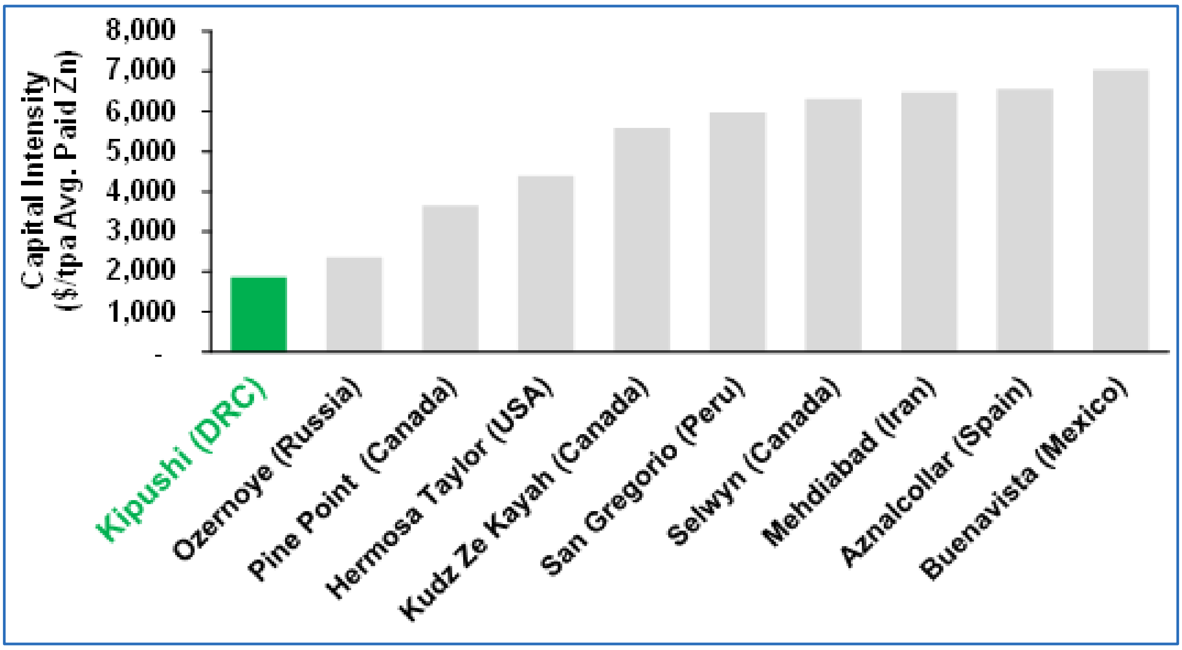


Wood Mackenzie, 2022

Kipushi's estimated low capital intensity relative to comparable 'probable' and 'base case' zinc projects identified by Wood Mackenzie is highlighted in Figure 1.14. The figure uses comparable projects as identified by Wood Mackenzie, based on public disclosure and information gathered in the process of Wood Mackenzie's research.

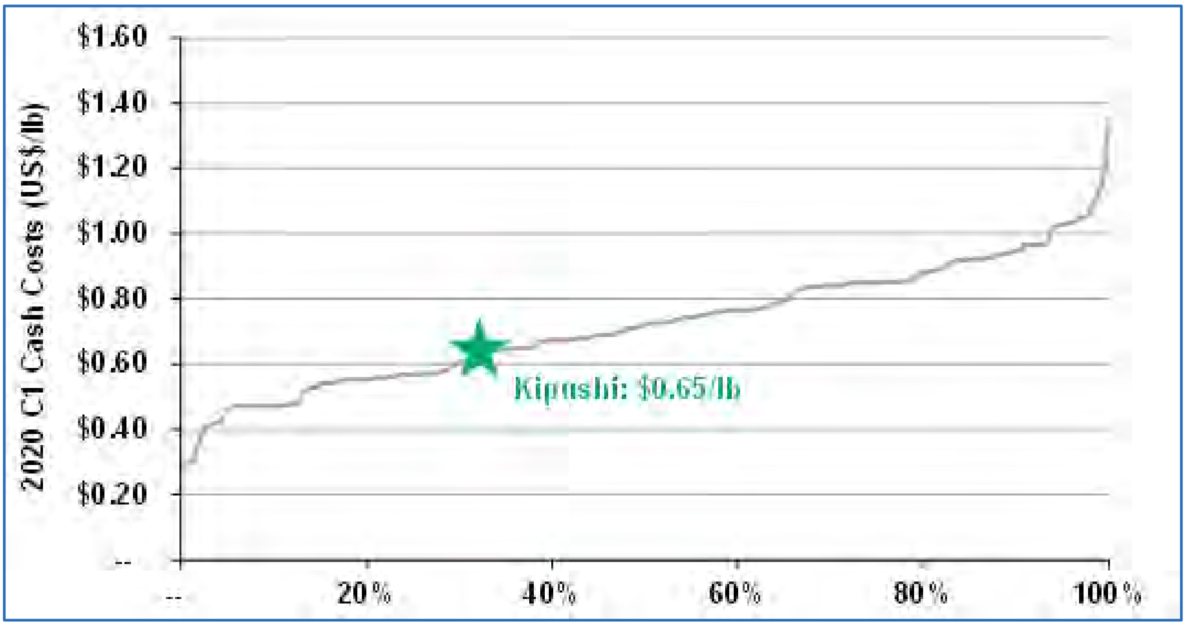
Based on data from Wood Mackenzie, life-of-mine average cash cost of US$0.65/lb of zinc is expected to rank Kipushi, once in production, in the second quartile of the 2020 cash cost curve for zinc producers globally (Figure 1.15). C1 cash costs reflect the direct cash costs of producing paid metal incorporating mining, processing and offsite realisation costs having made appropriate allowance for the co-product revenue streams.

Figure 1.14 Capital Intensity for Zinc Projects



Wood Mackenzie, 2022

Figure 1.15 2020 Expected C1 Cash Costs



Wood Mackenzie, 2022

1.21 Interpretation and Conclusions

1.21.1 Kipushi 2022 Feasibility Study

The Kipushi 2022 FS for the redevelopment of the Kipushi Mine is at a feasibility level of accuracy. It has identified a positive business case and it is recommended that the Kipushi Project should continue with early works programme for the project. There are a number of areas that need to be further examined and studied as the Kipushi Project's early works programme advances.

1.21.2 Mineral Resources

Mineral Resources for the Project have been estimated using core drill data, have been performed using industry best practices (CIM, 2003), and conform to the requirements of CIM Definition Standards (2014). MSA has checked the data used to construct the resource models, the methodology used to construct it, and has validated the resource models. MSA finds the Kipushi resource model to be suitable to support feasibility level mine planning.

Areas of uncertainty that may materially impact the Mineral Resource estimates include:

- Assumptions used to generate the data for consideration of reasonable prospects of eventual economic extraction for the Kipushi deposit.
  - Mining recovery could be lower, and dilution increased. Early stoping should be used to confirm mining method parameters for the Kipushi deposit in terms of costs, dilution, and mining recovery. Early development will also provide access to data and metallurgical samples at a bulk scale that cannot be collected at the scale of a drill sample.
- Commodity prices and exchange rates.
- Cut-off grades.

1.21.3 Mineral Reserve Estimation

Mineral Reserves for the Kipushi 2022 FS conform to the requirements of CIM Definition Standards (2014).

Areas of uncertainty that may impact the Mineral Reserve estimate include:

- Any changes to the resource model as a result of further definition drilling at the site.
- Commodity prices and exchange rates.

1.22 Recommendations

1.22.1 Further Assessment

The key areas for further studies/work are:

- On-going geotechnical drilling and logging will be required to increase the confidence in geotechnical data as the project develops.

- Ongoing geotechnical mapping should take place at regular intervals in the planned developments to verify the rock mass conditions determined and to assess the rock mass quality where there is currently little or no information. This will also allow for the identification of localised weak zones and potentially unstable wedges which should be appropriately supported.
- While the structural analysis provides an impression of the major joint sets across the project area, further geotechnical scanline mapping should be conducted regularly as mining commences to allow for the identification of low angle joints in the hanging wall, localised joint sets and for potential wedges and instabilities.
- Complete the preparatory work for the early works.
- Complete a trade-off study on alternative options for the backfilling strategy.
- Investigate potential sources of waste to meet the backfill shortfall.
- Define what infrastructure should be demolished to make the mine safe and operable.
- Optimise surface infrastructure layout.
- Review location of the new tailings dam.
- Investigate customer uptake for container transport.
- Investigate the optimal concentrate transport solution for bagging and bulk.
- Complete the regulatory Environmental and Social Impact Assessment (ESIA) and the Environmental and Social Management Plan (ESMP).
- Identify other permitting requirements.
- Prepare a detailed closure plan.
- It is recommended that additional boreholes should be drilled. The purpose of the boreholes is:
  - To verify the simulated water level drawdown north of the Petite Conglomerate.
  - To check the influence of mining on the well field.
  - To check the water quality downstream of the TSF in comparison with upstream water quality.
- The hydrogeological model is a water management tool that should be updated regularly (every two years) to incorporate.
- Ongoing monitoring data and to adjust the hydrogeological model as additional information becomes available.
- The hydrogeological model should be updated sooner if any additional inflows or other changes are observed during mining. Immediate action should be taken as soon as changes (i.e., should high permeable zones be encountered during mining) are observed.
- It is recommended that an unsaturated flow model should be constructed to investigate in more detail and to include the effect of evaporation. For this simulation, evaporation was included in the sense that a lower flux into the tailings was specified. An unsaturated flow model will give a specified flux value with higher certainty.

2 INTRODUCTION

2.1 Ivanhoe Mines Ltd.

Ivanhoe is a mineral exploration and development company, whose principal properties are located in Africa. The Ivanhoe strategy is to build a global, commodity-diversified mining and exploration company. Ivanhoe has focused on exploration within the Central African Copperbelt and the Bushveld Complex.

Ivanhoe currently has three key assets: (i) the Kamoa-Kakula Project; (ii) the Platreef Project, and (iii) the Kipushi Project. In addition, Ivanhoe holds interests in prospective mineral properties in the Democratic Republic of Congo (DRC) and South Africa.

Kipushi Holding (a wholly owned subsidiary of Ivanhoe) and Gécamines have a JV Agreement over the Kipushi Project. Kipushi Holding and Gécamines respectively own directly 68% and 32% of the Kipushi Project through KICO, the mining right holder of the Kipushi Project.

Ivanhoe's interest in KICO was acquired in November 2011 and includes mining right for copper, cobalt, zinc, silver, lead, and germanium as well as the underground workings and related infrastructure, inclusive of a series of vertical mine shafts.

2.2 Terms of Reference and Purpose of the Report

The Kipushi 2022 FS incorporates the Mineral Resource as reported in the March 2019 NI 43-101 Technical Report and these Mineral Resources remain unchanged and are valid and current for the purposes of this study.

The Kipushi 2022 FS is an Independent Technical Report on the Kipushi Project prepared for Ivanhoe Mines Ltd. (Ivanhoe) as part of the strategy for redevelopment of the Kipushi Project.

The following companies have undertaken work in preparation of the Kipushi 2022 FS:

- OreWin: Overall report preparation, underground mining, mineral processing, Mineral Reserve estimation, infrastructure, and financial model.
- The MSA Group Pty Ltd (MSA): Geology, Drillhole data validation, Sample preparation, Analysis and Security, and Mineral Resource estimation.
- SRK Consulting (South Africa) (Pty) Ltd (SRK): Mine geotechnical.
- MEC (Technical) Africa Pty Ltd (MEC): Mineral processing and infrastructure.

This Kipushi 2022 FS uses metric measurements except where otherwise noted. The currency used is US dollars ($).

2.3 Qualified Persons

The following people served as the Qualified Persons (QPs) as defined in National Instrument 43-101, Standards of Disclosure for Mineral Projects, and in compliance with Form 43-101F1:

- Bernard Peters, B. Eng. (Mining), FAusIMM (201743), employed by OreWin as Technical Director – Mining was responsible for: Sections 1.1–1.5, 1.10–1.12, 1.14–1.16, 1.18–1.22; Sections 2–5; Section 15; Sections 16.2–16.10; Sections 18.21–18.22; Section 19; Section 20; Sections 21.1–21.3, 21.5–21.6; Sections 22–24; Sections 25.2–25.3; Section 26.3; Section 27.
- Michael Robertson, BSc Eng (Mining Geology), MSc (Structural Geology), Pr.Sci.Nat SACNASP, FGSSA, MSEG, MSAIMM, employed by The MSA Group (Pty) Ltd as a Principal Consulting Geologist was responsible for: Sections 1.6–1.8, 1.21–1.22; Sections 2–3; Sections 6–12; Section 25.1; Section 26.2; Section 27.
- Jeremy Witley, BSc Hons (Mining Geology), MSc (Eng), Pr.Sci.Nat SACNASP, FGSSA, employed by The MSA Group (Pty) Ltd as a Principal Resource Consultant was responsible for: Sections 1.10, 1.21–1.22; Sections 2–3; Section 14; Section 25.1; Section 26.1; Section 27.
- William Joughin, FSAIMM (55634), employed by SRK Consulting (South Africa) (Pty) Ltd as Principal Consultant, was responsible for: Section 1.12; Section 2; Section 16.1; Section 27.
- John Edwards, F SAIMM (CP) (701196), BSc Hons (Mineral Processing Technology), ACSM, MBL, Director of METC Engineering PTY Ltd., was responsible for: Sections 1.9, 1.13, 1.17, 1.21–1.22; Section 2; Section 13; Section 17; Sections 18.1–18.20, 18.22; Section 21.4; Section 25.4; Section 26.4; Section 27.

2.4 Site Visits and Scope of Personal Inspection

Site visits were performed as follows:

- Mr Bernard Peters visited the Project from 1–3 June 2015, 11 September 2015, on 24 October 2016 and from 26–28 June 2017. The site visits included briefings from geology and exploration personnel, site inspections of potential areas for mining, plant and infrastructure, discussions with other QPs and review of the existing infrastructure and facilities in the local area around the Project site.
- Michael Robertson visited the Project from 20–23 February 2013 and again from 22–24 April 2013. The initial visit included a personal inspection of historical exploration records and drill core from the Project. During the subsequent visit, re-sampling of selected historical cores was undertaken as part of a data verification exercise.
- Jeremy Witley visited the Project from 8–11 September 2014, from 11–13 May 2015 and again from 13–15 November 2017.
- Mr William Joughin visited the project site from 19–22 May 2014, from 27–29 November 2017 and on 17 August 2018. The site visits included inspections of the drill core, underground visits to gain an impression of the ground conditions and discussions with the mine personnel on the local geology and previous mining activities conducted.

- John Edwards visited the Project from 24–28 October 2016 and reviewed the site relative to current process and infrastructure concepts with special reference to impediments, limitations, and opportunities. A basic level familiarisation was gained of the geology of the deposit, especially the Big Zinc, and the processing characteristics of the ore as understood from testwork was confirmed through an underground visit and viewing significant trays.

2.5 Effective Dates

The report has a number of effective dates, as follows:

- Effective date of the Report: 14 February 2022.
- Date of drillhole database close-out date for updated Mineral Resource estimate: 24 April 2018.
- Effective date of Mineral Resource update for mineralisation amenable to underground mining methods: 14 June 2018.
- Effective date of Mineral Reserves: 14 February 2022.

3 RELIANCE ON OTHER EXPERTS

The Qualified Persons (QPs), as authors of Kipushi 2022 Feasibility Report, have relied on, and believe there is a reasonable basis for this reliance, upon the following Other Expert reports as noted below. Individual QP responsibilities for the sections are listed on the Title Page.

The QPs, as authors of this report, have relied on the following sources of information in respect of mineral tenure and environmental matters pertaining to the Kipushi Project area.

3.1 Mineral Tenure

The QPs have not reviewed the mineral tenure, nor independently verified the legal status, ownership of the Kipushi Project area, underlying property agreements or permits. The QPs have fully relied upon and disclaim responsibility for, information derived from Kipushi Corporation SA (KICO) for this information through the following documents:

- KICO: report on the Kipushi Project Property Description and Location, March 2019.
- KICO: report on the Kipushi Project Property Description and Location, January 2018.
- A copy of the exploitation permit ('Certificat d'Exploitation') PE12434 dated 22 July 2011, issued by CAMI.
- A translation, from the original French into English, of the Kipushi Joint Venture Agreement No. 770/11068/SG/GC/2007 dated 14 February 2007 between La Générale des Carrières et des Mines (Gécamines) and Kipushi Resources International Limited (KRIL). Ivanhoe purchased the original KRIL 68% interest in the project.

This Kipushi 2022 FS has been prepared on the assumption that the Kipushi Project will prove lawfully accessible for exploration and mining activities.

3.2 Environmental and Permitting

- The QPs have obtained information regarding the environmental and work programme permitting status of the Kipushi Project through opinions and data supplied by KICO, and from information supplied by KICO staff. The QPs have fully relied on the following information provided by KICO in Section 4 and Section 20 Kipushi Environmental and Social Report, January 2018.
- Environmental Report on the Kipushi Zinc–Copper mine, Democratic Republic of Congo (DRC), by The Mineral Corporation, for Kipushi Resources International Limited (KRIL), 2007.
- Ivanhoe Mines Ltd., 2016: Kipushi Zinc Project – Preliminary Economic Assessment: unpublished letter prepared by representatives of Ivanhoe for OreWin, dated 12 May 2016.
- KICO: report on the Kipushi Project Property Description and Location, March 2019.
- KICO: report on the Kipushi Project Property Description and Location, January 2018.

3.3 Taxation and Royalties

The QPs have fully relied upon, and disclaim responsibility for, information supplied by Ivanhoe staff and experts retained by Ivanhoe for information relating to the status of the current royalties and taxation regime for the Project as follows:

- KICO: Email from KICO to OreWin on DRC Taxation for the Kipushi Project, November 2017.
- KPMG Services (Pty) Limited, 2016: Letter from M Saloojee, Z Ravat, and L Kiyombo to M Cloete, and M Bos regarding updated commentary on specific tax consequences applicable to an operating mine in the Democratic Republic of Congo, dated 1 March 2016.
- Ivanhoe Mines Ltd., 2016: Kipushi Zinc Project – Preliminary Economic Assessment: unpublished letter prepared by representatives of Ivanhoe for OreWin, dated 12 May 2016.
- KICO: report on the Kipushi Project Property Description and Location, March 2019.

This information was used in Sections 4 and 20 of the Kipushi 2022 FS.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

Kipushi town is situated approximately 30 km south-west of Lubumbashi, the capital of Haut Katanga Province. The geographical location of the Kipushi mine is 11°45'36" south and 27°14'13" east. The Kipushi Project is located in the DRC adjacent to the town of Kipushi, in the south-eastern part of the Haut-Katanga Province, adjacent to the border with Zambia.

The Kipushi Mine is a past-producing, high-grade underground zinc–copper mine in the Central African Copperbelt, which operated from 1924–1993, producing approximately 60 Mt at 11.03% Zn and 6.78% Cu. Additionally, over the period, from 1956 through to 1978, approximately 12,673 t of lead and 278 t of germanium was also produced (Ivanhoe, 2014). Mining at Kipushi began as an open pit operation, but by 1926 had become an underground mine, with workings stretching down to the 1,150 mL. In 1993, the mine was put on care and maintenance due to a combination of economic and political factors.

4.2 Project Ownership

Kipushi Holding, a company registered under the laws of Barbados which is indirectly owned by Ivanhoe, and Gécamines have a JV Agreement over the Kipushi Project. Kipushi Holding and Gécamines respectively own directly 68% and 32% of the Kipushi Project through KICO which holds the Exploitation Permit required for the implementation of this Project.

Kipushi Holding's interest in KICO was acquired in November 2011 and includes mining right for copper, cobalt, zinc, silver, lead, and germanium, as well as the underground workings and related infrastructure, inclusive of a series of vertical mine shafts.

The JV Agreement was signed on 14 February 2007 and established KICO for the exploration, development, production, and product marketing of the Kipushi Project. The JV Agreement document is Convention d'Association No. 770/11068/SG/GC/2007 (including appendices 1–5, A–F, and later amendments 1–6 to the JV Agreement) of 14 February 2007 between Gécamines and United Resources AG. United Resources AG was replaced by Kipushi Resources International Limited (KRIL) by amendment No.2 to the JV Agreement dated January 2009 and then Kipushi Holding purchased the KRIL 68% interest in the Project.

As announced on February 14, 2022, Kipushi Holding and Gécamines have agreed to commercial terms that will form the basis of a new Kipushi joint-venture agreement in order to establish a robust framework for the mutually beneficial operation of the Kipushi Mine. The new agreement remains subject to execution of definitive documentation.

4.3 Mineral Tenure

KICO holds the exclusive right to engage in mining activities within the perimeter of Exploitation Permit No. 12434 (PE12434), valid until 3 April 2024 and covering 07 mining squares and approximately505 ha. This permit is notably renewable under the terms of the DRC Mining Code, subject to the stability guarantee to which KICO is entitled to, notably pursuant to Article 276 of the 2002 Mining Code (Law No. 007/2002 dated 11 July 2002). The boundary coordinates of the Permit area are shown in Table 4.1.

Table 4.1 Boundary Coordinates for Permit Comprising the Kipushi Project (Coordinate system: Geographic WGS84)

Permit Number:	PE12434					
Type:	Exploitation Permit					
Area (Ha)[1]:	505.0					
Grant Date:	2/7/2011					
Expiry Date:	3/4/2024					
Point	Longitude			Latitude		
	Degree	Minute	Second	Degree	Minute	Second
1	27	14	0.00	–11	47	0.00
2*	27	13	49.86	–11	47	0.00
3*	27	13	40.75	–11	46	39.96
4*	27	13	39.32	–11	45	0.00
5	27	14	30.00	–11	45	0.00
6	27	14	30.00	–11	46	30.00
7	27	14	0.00	–11	46	30.00

[1] Surface area was calculated on the basis of the geographical coordinates set out in the mining tiles delivered to KICO by the CAMI and were not calculated on the basis of the regulatory size of the squares comprised within the Exploitation Permit.
* Exploitation Permit PE12434 is made up of cadastral squares (carrés), and any parts of these areas extending beyond the DRC borders are excluded from the licence.

Exploitation Permit No. 12434 resulted from the partial transfer of Exploitation Permit No. 481 previously held by Gécamines, was granted by Ministerial Order No. 0290/CAB.MIN/MINES/01/2011 dated 2 July 2011 and is evidenced by Exploitation Certificate No. CAMI/CE/6368/11 dated 22 July 2011, and granted KICO the exclusive right to perform exploration, development and exploitation works concerning silver, cobalt, copper, germanium, and zinc.

Exploitation Permit No. 12434 is still under a situation of Force Majeure duly approved by Decision No. CAMI/DG/FM/19/2012 dated 2 April 2012 until the Kipushi underground mine and its facilities have been refurbished.

CAMI challenged the continuation of the Force Majeure impacting Exploitation Permit No. 12434 in a letter received by KICO on 9 February 2021 and requested KICO to confirm, with evidence, the continuation of the approved Force Majeure.

KICO confirmed the continuation of Force Majeure in a letter dated 25 February 2021, clarifying its aim of finalising critical works by 24 March 2022 in order to subsequently note the end of the Force Majeure.

In order to examine this letter, CAMI requested, on 15 March 2021, the performance of an on-site survey by the Provincial Department of Mines and requested this Provincial Department of Mines to send CAMI minutes enabling CAMI to assess the accuracy of KICO's statements.

After on-site inspection performed by the Provincial Department of Mines, this latter concluded notably in its minutes dated of 06 May 2021 that it considered that the facts constituting the case of Force Majeure affecting the exercise and use of the mining right relating to Exploitation Permit No. 12434 continue and that the extension of the validity period can be considered.

There were no further development at CAMI's level since those minutes, to the best of Ivanhoe's knowledge so far.

The Zambian and DRC governments have both contracted FlexiCadastre (Spatial Dimension) to assist with the management of the mining rights of both states. This enables alignment regarding the management of mining rights on both sides of the border.

The boundaries of Exploitation Permit No. 12434, indicated in the Exploitation Certificate, cross the international border, as do some of the coordinates on the permits held as defined by CAMI. DRC permits are made up of cadastral squares (carrés) meaning the coordinates of the permit boundary (defined to the international border) and the permit blocks (defined by the cadastral squares) may not be coincidental.

As the DRC Mining Code does not apply in Zambia and therefore has no jurisdiction in Zambia, the right for KICO to mine stops at the international border, and any part of the Exploitation Permit area extending beyond the DRC borders are excluded from the Exploitation Permit.

The mineralisation at the Kipushi Project may extend, at depth, beyond the DRC border into Zambia. KICO does not have an agreement with the Zambian government which would permit it to explore for or exploit any Mineral Resources that may be in Zambia. The current Mineral Resource estimates presented for the Kipushi Project only make reference to those Mineral Resources which lie within the DRC.

4.4 Surface Rights

Exploitation Permit No. 12434 grants to KICO, without limitation, the exclusive right to perform within its perimeter the exploration, development, and exploitation works concerning the mineral substances identified in the relevant Exploitation Certificate.

In addition, pursuant to article 64 *bis* of the DRC Mining Code, Exploitation Permit No. 12434 enables KICO, without limitation, to:

- Enter into the exploitation perimeter to proceed to mining operations;
- Build the facilities and infrastructure necessary for mining exploitation;

- Use water and wood resources located within the mining perimeter for the needs of the mining exploitation subject to compliance with the norms defined in the relevant Environmental and Social Impact Assessment (ESIA) and Environmental and Social Management Plan (ESMP); and
- Proceed to the works of extension of the mine.

Pursuant to the legal principle whereby the accessories follow the main asset, the ownership of the assets and infrastructure in relation to exploitation of Exploitation Permit No. 12434 was transferred to KICO for the duration of Exploitation Permit No. 12434.

However, there are a number of exceptions, agreed between Kipushi Holding and Gécamines and to be interpreted restrictively, whereby Gécamines remained the owner, on the basis of specific land rights to be established in favour of Gécamines, of:

- The Old Concentrator of Kipushi (described in Appendix A of Amendment No. 3 to the JV Agreement);
- The New Concentrator of Kipushi (described in Appendix E of Amendment No. 3 to the JV Agreement);
- The Site of the Kipushi Tailings (Site des Rejets de Kipushi) corresponding to the site of storage of tailings, named basin No. 3 (Gécamines artificial deposits) described in Appendix F of Amendment No. 3 to the JV Agreement; and
- The Real Estate and Other Infrastructure of Kipushi (Immeubles et Autres Infrastructures de Kipushi) whose description is set out in Appendix D of Amendment No. 3 to the JV Agreement.

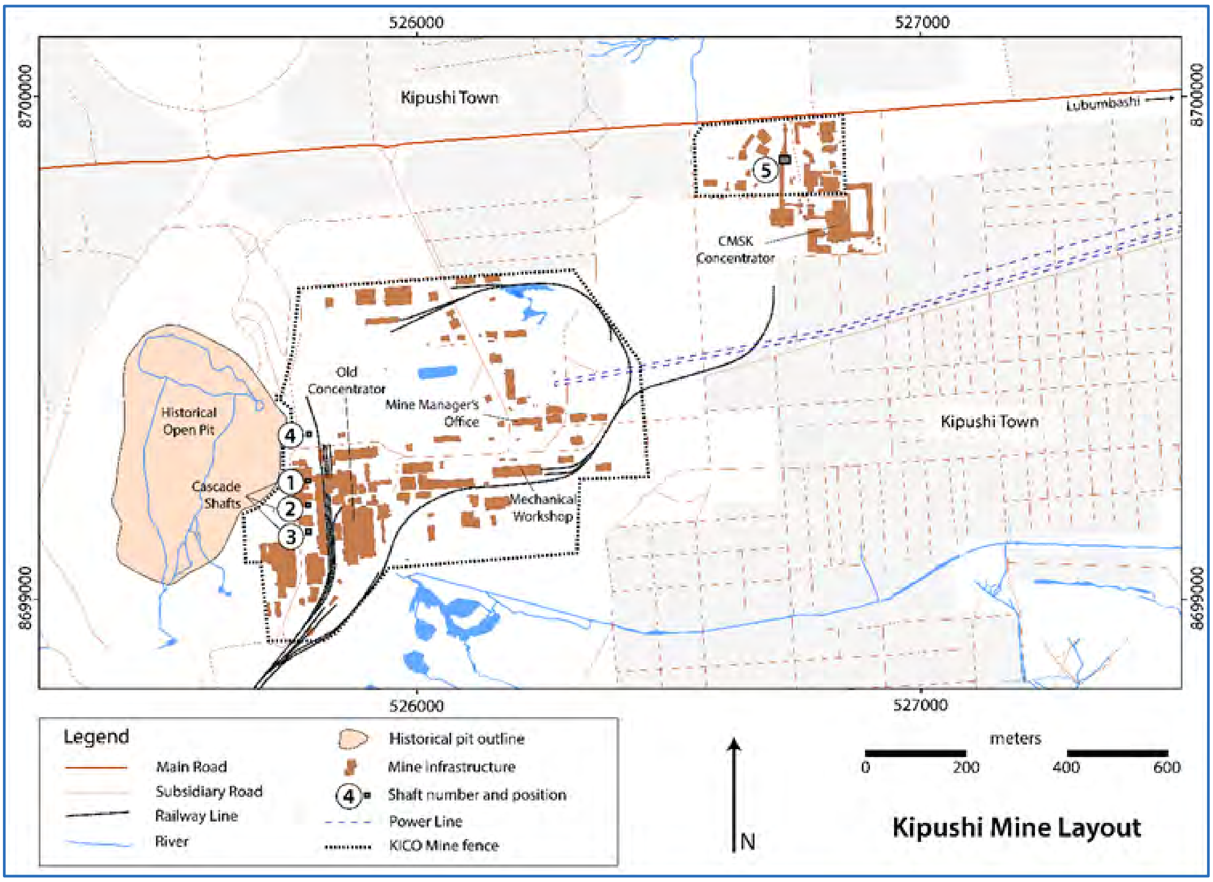
In addition, a number of assets defined as being the Rented Facilities and Equipment (Installations et Equipements Loués), described in Appendix C of Amendment No. 3 to the JV Agreement, are rented by Gécamines to KICO under a lease agreement that was the subject of a settlement agreement dated 14 June 2013. Gécamines, Kipushi Holding and KICO have agreed that an amendment to the above mentioned lease agreement should be discussed in good faith and would become an appendix to the revised JV Agreement to be negotiated and entered into.

Pursuant to the above-mentioned Appendix C, those Rented Facilities and Equipment include notably:

- Industrial facilities: High voltage station (Poste Haute Tension), pumping station of potable water, the Old Concentrator of Kipushi, the Cascade Mill, the Basin of tailings Katapula, two deposits of explosive products (dynamitières), building and facilities of KICO and SAT phone network; and
- A number of listed workshops required for the running of the mine, dewatering and warehouses.

The current Kipushi Mine layout is shown in Figure 4.1.

Figure 4.1 Kipushi Existing Mine Layout



Ivanhoe, 2015

4.5 Property Obligations and Agreements

Pursuant to the DRC Mining Code, the payment of surface right fees (droits superficiaires annuels) based on the number of cadastral squares held, in a timely manner, is generally required to maintain the validity of the mining rights held, KICO being entitled to a 10-year stability guarantee covering the rights attached to the Exploitation Permit, including without limitation, the right to the renewal of this mining right and tax and customs regimes in accordance with Article 276 of the 2002 Mining Code.

As Exploitation Permit No. 12434 is still under Force Majeure, KICO will pay the relevant surface rights fees for this mining right only when the Force Majeure will end.

4.6 Environmental Liabilities

The property covered notably by Exploitation Permit No. 12434 was the subject of an environmental audit by the Department in Charge of the Protection of the Mining Environment (DPEM) within the Ministry of Mines in August 2011.

DPEM subsequently granted Gécamines a release of its environmental obligations over the perimeter covered notably by Exploitation Permit No. 12434. KICO commissioned a summary environmental liabilities assessment study which was completed in August 2012 by Golder Associates. It serves as an environmental snapshot as to the state of the property when Kipushi Holding acquired the Kipushi Project in November 2011.

Pursuant to Article 405 alinea 3 of the applicable Mining Regulations, the holder that acquires its mining right by transfer (as KICO did from Gécamines), bears, for the transferor (Gécamines in this case), the environmental obligations vis-à-vis the State, unless the transferor obtained the certificate of release of its environmental obligations (which was obtained by Gécamines, as transferor of the Exploitation Permit in favour of KICO).

KICO is currently in the process of revising the approved Project Environmental and Social Impact Assessment (ESIA) and Environmental and Social Management Plan (ESMP).

4.7 Mining Legislation in the DRC

4.7.1 Mineral Property and Title

The following summary on mineral title is adapted from André-Dumont (2013) and from the DRC Mining Code.

The main legislation governing mining activities is the DRC Mining Code (Law No. 007/2002 dated 11 July 2002, as amended and completed by Law No.18-001 dated 9 March 2018), which is clarified by the Mining Regulations enacted by the Mining Regulations (Decree No. 038/2003 of 26 March 2003 as amended and completed by Decree No. 18/024 dated 8 June 2018 (Mining Regulations). These law and regulations incorporate environmental requirements.

All deposits of mineral substances within the territory of the DRC are state-owned. However, the holders of exploitation mining rights acquire the ownership of the products for sale (produits marchands) by virtue of their rights.

The Minister of Mines supervises, without limitation, the CAMI (DRC mining registry), the Departments of Mines and Geology and DPEM.

The main administrative entities in charge of regulating mining activities in the DRC as provided by the Mining Code and Mining Regulations are, without limitation, the following:

- The Prime Minister, who is notably responsible for enacting the Mining Regulations for the implementation of the Mining Code and declaring mineral substances as being a strategic mineral substance;
- The Prime Minister exercises his rights by decrees adopted in Council of Ministers, upon proposal of the Minister of Mines and, where appropriate, the relevant Ministers;
- The Minister of Mines, who has notably jurisdiction over the granting, refusal, and withdrawal of mining rights;

- The CAMI, is a public entity supervised by the Minister of Mines that is notably responsible for the management of the mining domain and mining rights. It conducts, without limitation, administrative proceedings concerning the application for, and registration of, mining rights, as well as the withdrawal and expiry of those rights;
- The Department of Mines is notably responsible for controlling and monitoring the performance of activities in relation to mines in accordance with legal and regulatory provisions in force;
- The DPEM is notably responsible in collaboration with the Congolese Agency for Environment, the national fund of promotion and social service and, where appropriate, any other relevant body of the State, for implementing the mining regulations concerning environment protection and performing the environmental examination of environmental and social impact studies, and environmental and social management plans. These administrations are also notably responsible for controlling and monitoring, without limitation, the obligations of the holders of mining rights concerning health and safety and the protection of environment in the sector of mines; and
- The Chief of the Provincial Department of Mines also has, without limitation, authority to control and monitor mining activities in Province.

Under the Mining Code, the mining rights are exploration permits, exploitation permits, small scale exploitation permits and tailings exploitation permits.

Foreign legal entities whose corporate purposes concern exclusively mining activities and that comply with DRC laws must elect domicile with an authorised DRC domestic mining and quarry agent (mandataire en mines et 56ere56y5656) and act through this intermediary. The mining or quarry agent acts on behalf of, and in the name of, the foreign legal entity with the mining authorities, mostly for the purposes of communication.

Foreign legal entities are eligible to hold only exploration mining rights. Foreign companies need not have a domestic partner, but a company that wishes to obtain an exploitation permit must transfer 10% (non-dilutable and free of any charge) of the shares in the share capital of the applicant company to the DRC State.

The Mining Code provides for a specific recourse system for mining right holders through three separate avenues that may be used to resolve mining disputes or threats over mining rights: administrative recourse, judicial recourse, or national or international arbitral recourse, depending on the nature of the dispute or threat.

The DRC is divided into mining cadastral grids using a WGS84 Geographic coordinate system outlined in the Mining Regulations. This grid defines uniform quadrangles, or cadastral squares, typically 84.955 ha in area, which can be selected as a "Perimeter" to a mining right. A perimeter under the Mining Code is in the form of a polygon composed of entire contiguous quadrangles subject to the limits relating to the borders of the National Territory and those relating to prohibited and protected reserves areas as set forth in the Mining Regulations.

Perimeters are exclusive and may not overlap subject to specific exceptions listed in the Mining Code and Mining Regulations. Perimeters are indicated on 1:200,000 scale maps that are maintained by the CAMI.

Within two months of issuance of an exploitation permit, the holder is expected to boundary mark the perimeter. The boundary marking (bornage) consists of placing a survey marker (borne) at each corner of the perimeter covered by the mining title and placing a permanent post (57ere57y) indicating the name of the holder, the number of the title and that of the identification of the survey marker.

4.7.2 Recent Amendment of the Legal Framework Governing Mining Activities and Local Procurement

When the 2002 Mining Code was introduced, the DRC Government indicated that after a 10-year period, a review would be undertaken.

Law No.18/001 dated 9 March 2018 amending and completing the 2002 Mining Code brought significant changes to the legal regime governing mining activities, including, without limitation, numerous issues, such as:

- Amendment of the stability guarantee set out by Article 276 of the 2002 Mining Code, with associated financial consequences for KICO.

KICO and Kipushi Holding, as well as the owners of the shares of Kipushi Holding, consider that in spite of the above mentioned amendment of the 2002 Mining Code and with regard to international law, they are entitled to the respect by DRC of the 10-year stability guarantee granted notably by Article 276 of the 2002 Mining Code, notably for all the rights attached to its mining rights, including, without limitation, the tax and customs regimes.

However, DRC applies to all mining companies, including KICO, since the entry into force of Law No.18/001, more stringent tax requirements adopted by Law No.18/001.

KICO and Kipushi Holding, as well as the owners of the shares of Kipushi Holding, consider that in requiring the immediate implementation of the more stringent tax requirements adopted by Law No.18/001, DRC breaches the clear and explicit stability guarantee that DRC undertook to respect in accordance with the 2002 Mining Code.

With regard to the current contrary interpretation of DRC, in spite of the requests made by KICO to have the stability guarantee respected by DRC and all its administrations during the stabilised period, KICO proceeds to the payment of the taxes required by DRC administrations, under duress and for the sole purpose of preventing, as far as possible, the damages that could result from sanctions imposed on KICO, while clarifying that such payments cannot be considered as a waiver to any of the right of KICO and in particular the stability guarantee it is entitled to:

- Increased tax and customs requirements reinforced by the breach by DRC of the stability guarantee it granted and to which KICO is entitled to.

Law No.18/001 inserted, without limitation, (i) a special tax on capital gains on the sale of shares whereby the tax administration is entitled to submit the capital gain on the sales of shares of an entity that has mining assets in the DRC, regardless of the actual territory where the transaction is entered into and (ii) a special tax on excess profits defined as the profit resulting from the increase of 25% of the commodities prices compared to those mentioned in the bankable feasibility study of the project.

Significant taxes that should not be applicable to KICO with regard to the stability guarantee it is entitled to are nevertheless applied by DRC administrations to KICO. They will increase the Kipushi Project's costs.

Also see comments in Section 4.7.6 concerning royalties:

- Increased importance of the commitments made vis-à-vis local communities on social and environmental aspects, the respect of the commitments made concerning social obligations in accordance with the schedule set out in the cahier des charges to be negotiated and entered into being a new condition to maintain the validity of the mining rights.

  Law No.18/001 also inserted an obligation to pay an annual contribution of 0.3% of the turnover for community development projects.

- Increased requirements concerning local procurement insofar as pursuant to the Mining Code, subcontractors, in the meaning of the Mining Code, must be DRC legal entities with Congolese financing ("congolais").

  Subject to further clarifications to be adopted, KICO understands from the recitals of Law No.18/001 that it means DRC companies having the majority of their share capital being directly held by Congolese individuals.

In addition, subcontracting activities, in the meaning of the Mining Code, must be performed in accordance with Law No.2017-01 dated 08 February 2017 determining the rules applicable to subcontracting in the private sector.

Pursuant to the 2017 Subcontracting law, subcontracting, in the meaning of the Subcontracting Law (which is distinct from the definition resulting from the Mining Code, as well as from the definition resulting from Mining Regulations), is an activity reserved to businesses with Congolese, financing, promoted by Congolese and having their head office in DRC. However, when there is non-availability or non-accessibility of the above expertise and subject to providing evidence to the relevant authority, the main contractor is authorised to enter into an agreement with any other Congolese or foreign business for a maximum duration of six months. The sectorial Minister or local authority must be informed previously.

Subcontracting, in the meaning of the Subcontracting Law, is limited to a maximum of 40% of the global value of a contract. In addition, the main contractor is not authorised to oblige the subcontractor, in the meaning of the Subcontracting Law, to totally prefinance the cost of the subcontracted operation or activity and must pay, before the beginning of the works, an advance payment covering at least 30% of the subcontracting contract. Any subcontracting above a threshold of approximately $50,000 (100,000,000 CDF) requires a public tendering process (appel d'offres). Fines for non-compliance with the Subcontracting Law are significant.

KICO therefore ensures that all its subcontractors, in the meaning of the Mining Code, strictly comply with the requirements of the Subcontracting Law.

These new rules increase the costs of the Project and could be considered as being contradictory, without limitation, with the stability guarantee to which KICO is entitled to and with Article 273 f of the Mining Code providing that mining companies holding mining rights are free to import goods, services as well as funds necessary to their activities subject to giving priority to Congolese businesses for all contracts in relation to the mining project, at equivalent conditions in terms of quantity, quality, price, delivery deadlines and payment.

KICO is nevertheless doing its best efforts to voluntarily ensure compliance with the new requirements, as well as with the relevant provisions of the JV Agreement and ongoing improvement in this respect to favour the development of local subcontractors, in the meaning of the Mining Code, as well as the selection of local subcontractors, in the meaning of the Subcontracting Law by its subcontractors, in the meaning of the Mining Code and Mining Regulations, that are, in KICO's understanding, main contractors (entrepreneurs principaux) in the meaning of the Subcontracting Law.

KICO also monitors the regulatory provisions adopted to ensure, as far as possible, adequate enforcement of the relevant legislative and regulatory requirements.

KICO however notes the ongoing significant confusion on the local procurement requirements applicable to mining companies and that the new authority governing subcontracting in the private sector (ARSP) claims that the agreements entered into between mining companies and their direct suppliers of goods and services constitute "subcontracting agreements" that should be governed by the Subcontracting Law.

This interpretation from ARSP would require, subject to limited exceptions or derogations, that all suppliers must be DRC companies, approved by ARSP as businesses eligible to perform subcontracting activities, including by having the majority of their share capital held by DRC individuals and managed by a majority of DRC individuals. This would also require burdensome tendering processes and possibly 30% advance payment to contractors.

ARSP could further claim a 1.2% payment on all goods and services provided to mining companies since October 2020.

If KICO was to apply the ARSP's interpretation, it could increase the costs of all procurement by approximately 1.2%, which would be a very significant amount in the context of the development and construction works and operation of the Project.

KICO's view is that notwithstanding the fact that the legality of ARSP and of its expectations could be challenged, they should concern only subcontracting agreements, as defined by the Subcontracting Law, entered into between main contractors, as defined by the Subcontracting Law and subcontractors, as defined by the Subcontracting Law and therefore not concern the provision of goods to KICO insofar as subcontracting, as defined by the Subcontracting Law, only concerns services contracts (contrats d'entreprise) and not sale agreements and should not concern the relationships between KICO and its direct services contractors, whereby KICO acts as a client (maitre d'ouvrage), in the meaning of the Subcontracting Law, with its main contractors, that are not governed by the Subcontracting Law.

Also, KICO's understanding of applicable law is, in particular, that ARSP is not competent to verify the enforcement of the Mining Code or Mining Regulations.

The Subcontracting Law also includes requirements that could be claimed as being applicable to all companies, such as, for instance, an obligation to publish each year the list of the subcontractors, in the meaning of the Subcontracting Law, and to implement, within the companies, a training policy enabling Congolese to acquire the technicity and qualification required for the performance of some activities.

In spite of the fact that the applicability of those requirements to KICO is, in its view, challengeable, with regard to the purpose of the Subcontracting Law, KICO voluntary performs, as far as possible, those obligations in spite of the numerous uncertainties resulting from a lack clarity of the implementing regulations and monitors the regulations that could clarify such requirements in the future.

KICO is also considering the actions to be put in place to address, as far as possible, ARSP's expectations and working to coordinate and align, as far as possible, its approach with FEC (Fédération des Entreprises du Congo), KICO being a member of FEC.

Uncertainties resulting from a lack of clarity of the implementing regulations include:

- Increased requirements on local processing and transformation of exploited mineral substances;
- More stringent rules applicable to the transfer of interests in DRC projects;
- Increased obligation to repatriate in DRC sale proceeds (when in production); and
- The obligation to transfer an additional 5% of the shares in the share capital of the company upon each renewal of the exploitation permits.

Among the risks resulting from this new legal and regulatory framework, one can also mention, without limitation, the risks associated to:

- The minerals substances declared as being strategic substances that can be changed anytime by a decree from the Prime Minister deliberated in Council of Ministers, upon an opinion from the relevant sectorial Ministers, the royalty applicable to such strategic substances being 10%.

  Pursuant to Decree No.18/042 dated 24 November 2018, cobalt, germanium, and colombo-tantalite "coltan" were declared as being 'strategic mineral substances'; and
- The mining products for sale that must be compliant with the nomenclature set out by the applicable regulations that could possibly change, from time to time, by DRC Government. KICO notes in this respect that zinc concentrate is not explicitly mentioned in the current version of the applicable nomenclature. This issue will need to be addressed, in a timely manner, for the purpose of the implementation of the Project.

There are also a number of new requirements, such as the obligation to build a building for the registered office, the obligation to have a share capital reaching at least 40% of the required financial resources or distinct mines that remain unclear. However, subject to contrary interpretations from the local courts and administrations, KICO's view is that those new requirements should not currently apply to KICO.

KICO, Kipushi Holding and the owners of the shares of Kipushi Holding consider that KICO should be protected against most adverse changes impacting the rights attached to its mining right, including the tax regime applicable to the mining right with regard to the 10-year stability guarantee KICO is entitled to in accordance with Article 276 of the 2002 Mining Code in particular. They nevertheless note the current contrary interpretation adopted by DRC and its administrations.

4.7.3 Exploitation Permit

Pursuant to the Mining Code, exploitation permits are valid for 25 years, renewable for periods that do not exceed 15 years until the end of the mine's life, if conditions laid out in the Mining Code are met.

Granting of an exploitation permit is dependent on a number of conditions that are defined in the Mining Code, including:

1. Demonstration of the existence of an economically exploitable deposit by presenting a feasibility study compliant with the requirements of the laws of the DRC, accompanied by a technical framework plan for the development, construction, and exploitation work for the mine.
2. Demonstration of the existence of the financial resources required for the carrying out of the holder's project, according to a financing plan for the development, construction, and exploitation work for the mine, as well as the rehabilitation plan for the site when the mine will be closed. This plan specifies each type of financing, the sources of financing considered and justification of their probable availability. In all cases, the share capital brought by the applicant cannot be less than 40% of the said resources.
3. Obtain in advance the approval of the project's Environmental and Social Impact Assessment (ESIA) and Environmental and Social Management Plan (ESMP).
4. Transfer to the DRC State 10% of the shares constituting the share capital of the company applying for the exploitation permit. These shares are free of all charges and cannot be diluted.
5. Creation, upon each transformation, in the framework of a distinct mine or a distinct mining exploitation project, an affiliated company in which the applicant company holds at least 51% of the shares.
6. Filing of an undertaking deed whereby the holder undertakes to comply with the cahier des charges defining the social responsibility *vis-à-vis* the local communities affected by the project's activities.
7. Having complied with the obligations to maintain the validity of the permit set out in Articles 196, 197, 198 and 199 of the Mining Code, by presenting:
    - The evidence that the certificate of the beginning of works was duly delivered by the Cadastre Minier; and
    - The evidence of payment of the surface right fees payable per squares (carrés) and of the tax on the surface area of mining concessions.

8. Providing the evidence of the capacity to treat (traiter) and transform the mineral substances in the DRC and filing an undertaking deed to treat and transform these substances within the Congolese territory.

The exploitation permit, as defined in the Mining Code, grants to its holder the exclusive right to carry out, within the perimeter over which it is established, and during its period of validity, exploration, development, construction and exploitation works in connection with the mineral substances for which the exploitation permit was granted, and associated substances if the holder has applied for an extension.

In addition, it entitles, without restriction, the holder to:

1. Enter within the exploitation perimeter to proceed with mining operations;
2. Build the facilities and infrastructure required for mining exploitation;
3. Use the water and wood resources located within the mining Perimeter for the needs of the mining exploitation, in complying with the norms defined in the ESIA and the ESMP;
4. Dispose (disposer), transport and freely market this products for sale originating from within the exploitation perimeter;
5. Proceed with concentration, metallurgical or technical treatment operations, as well as the transformation of the mineral substances extracted from the deposit within the exploitation Perimeter; and
6. Proceed to works of extension of the mine.

For renewal purposes under the Mining Code, a holder must, in addition to supplying proof of payment of the filing costs for an exploitation permit and without limitation, show that the holder has:

- Not breached the holder's obligations to maintain the validity of the exploitation permit set out in Articles 196 to 199 of the Mining Code;
- Presented a new feasibility study in accordance with the laws and regulations of the DRC demonstrating the existence of exploitable reserves;
- Demonstrated the existence of the financial resources required to continue to carry out his project in accordance with the financing and mine exploitation work plan, as well as the rehabilitation plan for the site when the mine will be closed. This plan specifies each type of financing considered and the justification of its probable availability;
- Obtained the approval of the update of the ESIA and ESMP;
- Undertaken to actively carry on with his exploitation;
- Demonstrated the entry of the project in its phase of profitability;
- Demonstrated the regular and uninterrupted development (mise en valeur) of the project;
- Transferred to the State, upon each renewal, 5% of the shares in the share capital of the company, in addition to those previously transferred;
- Not breached its tax, non-tax (parafiscal) and customs obligations; and

- Undertaken to comply with the cahier des charges defining the social responsibility vis-à-vis the local communities affected by the project's activities.

Under the Mining Code, a mining rights holder must pay in a timely manner a levy on the total surface area of his mining title (Article 238 of the Mining Code). Levies are defined on a per hectare basis, and increase on a sliding scale for each year that the mining right is held, until the third year, after which the rate remains constant. In this Report, this levy is referred to as a "tax on the area of mining concession" (taxe sur la superficie sur les concessions minières).

An additional duty (Article 199 of the 2002 Mining Code) (droit superficiaires annuel par carré), meant to cover service and management costs of the Cadastre Minier and the Ministry of Mines, and payable annually to the CAMI before 31 March, is levied on the number of squares held by a title holder. Different levels of duties are levied depending on the number of years a mining title is held, and whether the mining right is an exploration or exploitation mining right. In this Report, this tax is referred to as "annual superficiary rights" or surface right fee.

4.7.4 Sale of Mining Products

Pursuant to Article 85 the Mining Code, the trading of mining products which originate from the exploitation permit is "free", meaning that the holder of an exploitation permit may sell its products to customers of its choice, at "prices freely negotiated".

However, pursuant to Article 108 *octies* of the Mining Code, the trading of the mining products that originate from exploitation perimeters must be done in accordance with the laws and regulations in force in DRC. This provision also specifies that the holder of an exploitation permit may sell its products to clients of its choice at fair price with regard to market conditions.

However, in the case of a local sale, it can only sell its products to a legal entity exercising mining activity or to manufactures having a link with mining activity. Mining products for sale must be compliant with the nomenclature set out by the relevant regulations.

A specific authorisation, governed by the Mining Code and Regulations, is required for exporting unprocessed ores (minerais à l'état brut) for processing outside the DRC.

4.7.5 Surface Rights Title

The following summary on surface rights title is adapted from André-Dumont (2008, 2011) and from the Mining Code.

The soil is the exclusive, non-transferable, and lasting ownership of the DRC State (Law No. 73-021 dated 20 July 1973, as amended by Law No. 80-008 dated 18 July 1980). However, the DRC State can grant surface rights to private or public parties.

Surface rights are distinguished from mining rights, since surface rights do not entail the right to exploit minerals or precious stones. Conversely, a mining right does not entail any surface occupation right over the surface, other than that required for the operation.

The Mining Code provides that subject to the potential rights of third parties over the relevant soil, the holder of an exploitation mining right has, with the authorisation of the Governor of the relevant Province, after opinion from the relevant department of the Administration of Mines notably within the perimeter of the mining right, the right to occupy the parcels of land required for its activities and the associated industries, including the construction of industrial facilities, dwellings and facilities with a social purpose, to use underground water, the water from non-navigable, non-floatable watercourses, notably to establish, in the context of the concession of a waterfall, an hydroelectric power plant aimed at satisfying the energy needs of the mine, to dig canals and channels, and establish means of communication and transport of any type.

Any occupation of land that deprives the beneficiaries of land use and any modification rendering the land unfit for cultivation, entails, for the holder of mining rights, at the request of the beneficiaries of land use and at their convenience, the obligation to pay a fair compensation corresponding either to the rent or to the value of the land when it is occupied, increased by the half. The mining rights holder must also compensate the damages caused by its works that it performs in the context of its mining activities, even when such works were authorised.

Finally, in the event of displacement of populations, the holder of the mining right must previously proceed to the compensation and resettlement of the concerned populations.

4.7.6 Royalties

A company holding an exploitation permit is subject to mining royalties.

Pursuant to the 2002 Mining Code, the mining royalty is due upon the sale of the product and is calculated at 2% of the price received of non-ferrous metals sold less the costs of transport, analysis concerning quality control of the commercial product for sale, insurance and marketing costs relating to the sale transaction.

The holder of the exploitation permits should benefit from a tax credit equal to a third of the mining royalties paid on products sold to a transformation entity located in the National Territory. Mining royalties paid may be deducted for income tax purposes.

Amendments to the 2002 Mining Code were nevertheless adopted by the above-mentioned Law No.18/001 dated 9 March 2018.

Pursuant to Law No.18/001, the holder of the exploitation permit is subject to a mining royalty whose basis (assiette) is calculated on the basis of the gross commercial value and must pay this royalty on any product for sale as from the date of beginning of the effective exploitation.

The mining royalty is calculated and payable at the moment of the exit of the extraction site or of the treatment facilities for expedition. The rate of the royalty is increased to 3.5% instead of 2% for non-ferrous and/or base metals and 10% for strategic substances.

At the date of this Report, the zinc concentrates that KICO intends to sale and export is not listed among the strategic mineral substances. Only germanium covered by the Exploitation Permit was declared as being a strategic mineral substance by Decree No. 18/042 dated 24 November 2018.

Pursuant notably to Article 276 of the 2002 Mining Code and insofar as KICO holds mining rights that were valid when Law No.18/001 entered into force, KICO, Kipushi Holding and the owner of the shares of Kipushi Holding consider that KICO is entitled to the 10-year stability guarantee covering the tax regime applicable to its mining rights for the royalties payable in relation to the products from these mining rights.

KICO nevertheless notes the contrary interpretation from DRC and its administrations on similar issues and the opinion from KICO, Kipushi Holding and the owners of the shares in Kipushi Holding is that in the event DRC would impose KICO the forced enforcement of the above mentioned more stringent tax rules resulting from Law No.18/001 for products covered by the stability guarantee and within the stabilised period, this would constitute a breach to the stability guarantee to which KICO is entitled to.

In addition to this mining royalty governed by the Mining Code, Gécamines is entitled to a contractual royalty pursuant to the JV Agreement whereby in consideration of the consumption of ores, KICO shall pay Gécamines quarterly, in the form of royalties, 2.5% of the Net Turnover. This is further specified in the JV Agreement.

4.7.7 VAT Exoneration

Holders of mining rights are normally entitled to exoneration for import duties and import VAT for all materials and equipment imported for construction of a mine and related infrastructure.

4.7.8 Environmental Regulations

The following summary on environmental regulations is adapted from André-Dumont (2008, 2011) and from the Mining Code.

All exploration, mining and quarrying operations must have an approved environmental plan and the holders of the right to conduct such operations are responsible for compliance with the rehabilitation requirements stipulated in the plan.

When applying for an exploitation permit, a company must complete an ESIA to be filed, together with the ESMP to be approved by the relevant authorities.

On approval, the applicant must provide a financial guarantee for rehabilitation. This guarantee can be provided by means of a bank guarantee. Funds posted as guarantee are not at the disposal of the DPEM and are to be used for the rehabilitation of a mining site.

As previously indicated, KICO is in the process of updating its approved Environmental Social Impact Assessment (ESIA) and Project Environmental and Social Management Plan (ESMP).

Environmental obligations for conversion of an exploration permit to an exploitation permit under the Mining Code require the preparation of an ESIA and an ESMP.

The holder of a mining right submitted to an ESIA of the Project must revise its initially approved ESIA and ESMP and to sign them:

- Every five years;
- When its rights are renewed;
- When changes in the mining activities justify an amendment of the project ESIA; and
- When a control and/or monitoring report demonstrates that the mitigation and rehabilitation measures planned in its ESMP are no longer adapted and that there is a significant risk of adverse impact for the environment.

The Mining Regulations also require an environmental and social audit every two-year period as from the date of approval of the initial ESIA. The report of the last bi-annual environmental audit concerning the Exploitation Permit was filed to DPEM on 17 July 2020.

Breaches with environmental obligations can lead to significant sanctions, including suspension of mining activities and confiscation of the financial guarantees, subject to strict compliance with the formalism and proceedings described in the relevant laws and regulations.

Upon mine closure, shafts must be filled, covered, or enclosed. After a closure environmental audit and an in-situ audit by the DPEM together with the Environment Congolese Agency and the national fund of promotion and social service, a certificate of release of environmental obligations can be obtained.

Mining Regulations also mention exploitation permits for 1A classified facilities which depends on the payment of a tax.

The Mining Code finally provides that clearing permits must finally be obtained before any clearing is performed, if any.

In accordance with the above-mentioned obligations, KICO is in the process of revising the approved Project Environmental and Social Impact Assessment (ESIA) and Environmental and Social Management Plan (ESMP).

4.7.9 Surface Rights

Surface rights (which are distinct from mining rights) for the Kipushi Project are held by Gécamines. KICO, as holder of the exploitation permit, has, subject to the applicable approvals, authorisations and the payment of any requisite compensation, the right to occupy that portion of the surface as is within the exploitation permit area and which is necessary for mining and associated industrial activities. This includes the construction of industrial plants and the establishment of a means of communication and transport.

In order to access the surface infrastructure, KICO has entered into a rental contract with an affiliate of Gécamines, for the exclusive right to use the surface infrastructure held by Gécamines. Pursuant to which KICO will be required to pay rental fees of $100,000 per month. However, until the Force Majeure condition has been lifted KICO is paying rental fees of $30,000 per month to lease the areas required for its operations.

Subject to the observations detailed in Section 4.4, KICO, as holder of the Exploitation Permit, has, subject to the applicable approvals, authorisations and the payment of any requisite compensation, where appropriate, the right to occupy that portion of the surface as is within the Exploitation Permit area and which is necessary for mining and associated industrial activities, including the construction of industrial plants and the establishment of means of communication and transport.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

Information in this section is largely sourced from Ivanhoe (2015).

5.1 Accessibility

The town of Kipushi and the Kipushi Mine are located adjacent to the international border with Zambia, approximately 30 km south-west of Lubumbashi, the capital of Haut-Katanga Province and nearest major urban centre. The closest public airport to the Kipushi Project is at Lubumbashi where daily domestic, regional, and international flights are scheduled.

5.2 Climate and Physiography

The Lubumbashi region is characterised by a humid subtropical climate with warm rainy summers and mild dry winters. Most rainfall occurs during summer and early autumn (November to April) with an annual average rainfall of 1,208 mm. Average annual maximum and minimum temperatures are 28°C and 14°C respectively.

Historical mining operations at the Kipushi Project operated year-round, and it is expected that any future mining activities at the Kipushi Project would also be operated on a year-round basis.

The Katanga region occupies a high plateau covered largely by Miombo (Brachystegia sp.) woodland and savannah. Kipushi lies at approximately 1,350 mamsl with a gently undulating topography with shallow valleys created by small streams. The international border with Zambia is defined by a watershed. On the DRC side a prominent drainage basin has developed, flowing to the east into the Kafubu River.

5.3 Local Resources and Infrastructure

The town of Kipushi lies adjacent to the Kipushi Project area and near the mine's infrastructure and underground access.

Although the town of Kipushi is theoretically administered independently of the mine, La Générale des Carrières et des Mines (Gécamines) runs the schools, hospital, and water supply (Kelly et al., 2012). Over the considerable time that the mine has been in operation, the town and mine have become interlinked with operations very proximal to habitations.

Prior to the suspension of mining operations in 1993, the mine was the largest employer of the local population, with many of these people still living in the area. Following the suspension of mining operations, a number of mine personnel have been retained for care and maintenance operations and to keep the mine secure. As of 31 December 2014, KICO still employed approximately 400 people.

A link with the rail system in neighbouring Zambia provides access to the ports of Dar es Salaam in Tanzania, Maputo in Mozambique, and Durban in South Africa. Presently however, much of the product from mines in the Haut-Katanga Province is transported by road.

KICO has a significant amount of underground infrastructure at the Kipushi Project, including a series of vertical mine shafts, to various depths, associated head frames, and accompanying underground mine excavations. The newest shaft (Shaft 5) is 8 m in diameter and 1,240 m deep with the lowest operating level of 1,150 mL. It provides the primary access to the lower levels of the mine, including the Big Zinc. It has three independent friction hoists, and all compartments remain operational. The condition of the facility is fair but will require a refurbishment programme to bring the whole mine shaft to a working standard. Shaft 5 is approximately 1.5 km from the main mining area. A series of cross-cuts and ventilation infrastructure are still in working condition. The underground infrastructure also includes a series of pumps to manage the influx of water into the mine. Until 2011 the pumps de-watered down to a pump station at 1,210 mL. This station failed in 2011 and water level rose to 862 mL at its peak. Since Ivanhoe has assumed responsibility for ongoing rehabilitation and pumping, the water level was lowered and stabilised at approximately 1,300 mL on the Cascades Shaft 1 Tertiary, allowing underground diamond drilling from the 1,272 mL hanging wall drive. The underground infrastructure which has been exposed since dewatering, is in relatively good order.

The following describes the status of the current underground refurbishment works:

- Shaft Works:
  - All three Hoists (winders) have been upgraded and commissioned.
  - Shaft buntons and guides have been inspected and repaired to allow hoists to operate. There is a replacement programme underway to replace a number of the existing shaft bunton sets.
  - New shaft conveyances have been procured and have been installed except for the new cage which will be installed on completion of the bunton replacement programme. The older cage is fully operational.
  - Headframe loadout bin and vibratory feeder onto the P4 conveyor to the stockpile is fully operational.
  - The Cascade Hoisting (Winding) system is being upgraded.
  - P2 hoist has been replaced and commissioned.
  - P15 hoist is being stripped and new components installed in September 2021.
  - P2 Bis hoist has been procured and will be installed early 2022.
- Crushing Station:
  - The crusher has been replaced and commissioned along with the feed conveyors from the crusher to the Silos.
  - Silos and the feed to the shaft skip loading facility are fully commissioned and operational.
- A workshop at 1,150 mL is fully operational equipped with new 5 t and 20 t cranes.
- Pump Stations:
  - All five 3.5 MW-12 stage pumps at the 1,200 mL pump station have been replaced and commissioned.

- A new pump station with three 3.2 MW-8 stage pumps at the 850 mL has been constructed and commissioned to replace the old cascade pumping system. The latter being kept for emergency use.

The Kipushi Project includes surface mining and processing infrastructure, offices, workshops, and a connection to the national power grid. The property also hosts historical infrastructure, such as a mineral processing/beneficiation plant, that will not be further utilised. Electricity is supplied by the DRC state power company, Société Nationale d'Electricité (SNEL), from two transmission lines from Lubumbashi. Pylons are in place for a third line. Gécamines owns most of the surface infrastructure that it leases to KICO pursuant to the JV Agreement and a lease agreement.

The bulk of the Mineral Resources, and exploration potential, lie adjacent to or below the 1,150 mL main haulage level, which can be accessed from Shaft 5. This shaft has provided the main access underground since suspension of production and remains operational since the completion of dewatering at the end of 2013. Hanging wall drill stations are present on 1,132 mL and 1,272 mL, and an underground decline is developed in the footwall to approximately 1,330 mL. The re-establishment of operations at the Kipushi Project would require refurbishment of underground access via Shaft 5, and construction of new ore processing and waste disposal facilities. Metallurgical testwork conducted indicates that the raw water recovered from mine dewatering is not appropriate for use in the flotation circuit. Therefore, potable water would be utilised as process water make-up. Provisions, however, have been made for using raw water for process make up water if required.

6 HISTORY

Prior to formal mining at Kipushi, the site was the subject of artisanal mining by means of pits and galleries. The artisanal workings were visited in August 1899 by an exploration mission of the Tanganyika Concessions Ltd led by George Grey and were first named Kaponda after the local chieftain and later Kipushi in reference to the nearby river and village (Heijlen et al., 2008).

A Belgian company, Union Minière du Haut Katanga (UMHK) started prospecting in the area in 1922 and commenced production in 1924. UMHK reportedly operated on a more or less uninterrupted basis for 42 years, initially by open pit until 1926 and subsequently by the underground methods of sub-level caving and sub-level stoping. The mine was originally known as the Prince Leopold Mine. In 1966, with the formation of the State-owned mining company La Générale des Carrières et des Mines (Gécamines), the renamed Kipushi Mine was nationalised.

Mining of the Fault Zone and Copper Nord Riche zone continued under Gécamines management until 1993, reaching 1,150 mL, when, due to a lack of hard currency to purchase supplies and spares, the mine was put on care-and-maintenance.

Following an open bidding process in October 2006, United Resources AG commenced negotiations with Gécamines, resulting in the February 2007 Kipushi joint venture Agreement (JV Agreement) and the creation of the joint venture company, Kipushi Corporation SA (KICO). In May 2018, United Resources AG novated the Kipushi JV Agreement to the Kipushi Vendor via a novation act, with the Kipushi Vendor replacing United Resources AG as a party to the Kipushi JV Agreement.

In November 2011, Ivanhoe acquired 68% of the issued share capital of KICO through Kipushi Holding, from the Kipushi Vendor, the result of which the Kipushi Vendor transferred all of its rights and obligations under the Kipushi JV Agreement to Ivanhoe.

Prior to closure, the Kipushi deposit had largely been mined from surface down to approximately the 1,150 mL. The 1996 WGM report (Ehrlich, 1996) records Gécamines production from 1926–1993 as approximately 60 Mt at 11.03% Zn for 6.6 Mt of zinc and 6.78% Cu for 4.1 Mt of copper. Between 1956 and 1978, 12,673 t of lead and approximately 278 t of germanium in concentrate were produced. Historically, a zinc and copper concentrate were produced from sulfide feed.

In addition to the recorded production of copper, zinc, lead, and germanium, historical Gécamines mine-level plans for Kipushi also reported the presence of precious metals. There is no formal record of gold and silver production; the mine's concentrate was shipped to Belgium and any recovery of precious metals was not disclosed during the colonial era.

Historical resource estimates below 1,150 mL were established through Gécamines' diamond drilling and limited underground sampling.

Three historical resource estimates have been prepared on the Kipushi Project. These were undertaken by Gécamines (1994), Watts, Griffis and McOuat Limited (WGM) (1996), and Techpro Mining and Metallurgy (Techpro) (1997). In addition, Zinc Corporation of South Africa (Zincor) is reported to have made an estimate in 2001 using proprietary geological modelling software (Kelly et al., 2012). All were based on Gécamines' drilling and production information, and utilised Gécamines' historical cut-off grades.

A first-time Mineral Resource estimate was prepared by The MSA Group (Pty) Ltd (MSA) for the Kipushi Project in 2006, and the estimate has now been updated in 2019.

Preliminary Economic Assessments (PEA) on the Kipushi Project were prepared in 2016 (Peters et al., 2016). The Kipushi 2016 PEA examined a 1.1 Mtpa production rate a similar mining method, dense media separation (DMS) processing and rail transport options for concentrate.

The Kipushi Mineral Resource Estimate was released in a Technical Report in March 2016, this was followed by the Kipushi 2017 Pre-feasibility Study and updated in the Kipushi 2019 Resource Update.

The previous Technical Report was the Kipushi 2019 Resource Update which presented the results of additional diamond drilling completed by KICO from May 2017 to November 2017 and an updated mineral resource estimate for the Kipushi Project.

As announced on February 14, 2022, Kipushi Holding and Gécamines have agreed to commercial terms that will form the basis of a new Kipushi joint-venture agreement in order to establish a robust framework for the mutually beneficial operation of the Kipushi Mine. The new agreement remains subject to execution of definitive documentation.

7 GEOLOGICAL SETTING AND MINERALISATION

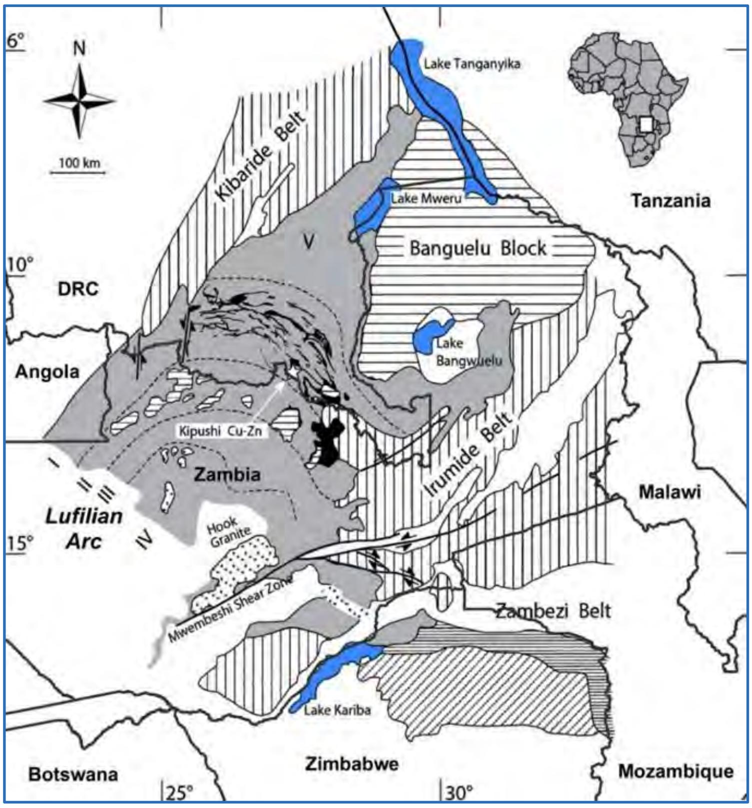
This section has not been changed from the Kipushi 2019 Resource Update and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Kipushi 2019 Resource Update.

The following review of the geological setting of the Kipushi Project has been compiled from published literature as cited and as referenced in this Report, together with geological knowledge gained by Kipushi Corporation SA (KICO) during the course of its underground drilling programme. A reinterpretation of the geology has recently been published in Economic Geology (Turner et al., 2018), which forms the basis for many of the updates to the geology section.

7.1 Regional Geology

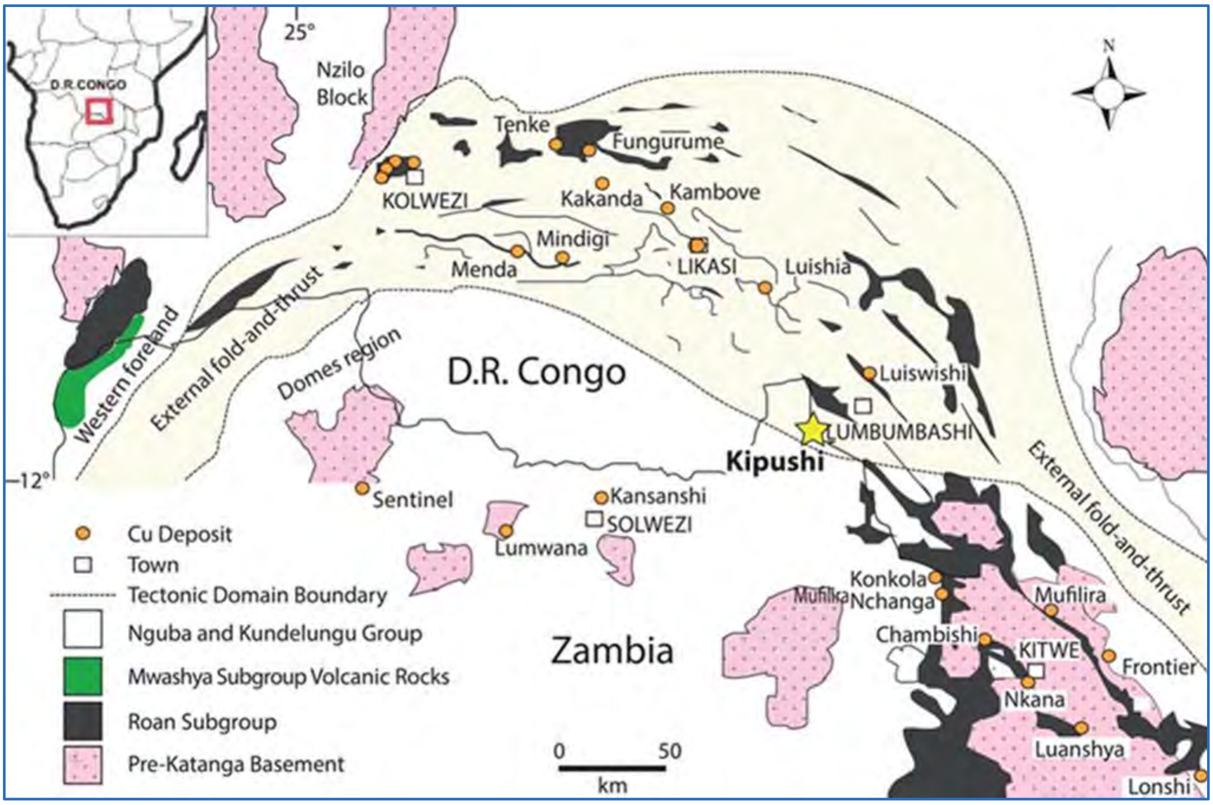
Kipushi is located within the Central African Copperbelt, a northerly convex arc extending approximately 500 km from north central Zambia through the southern part of the Democratic Republic of Congo (DRC) into Angola (Figure 7.1). The Central African Copperbelt constitutes a metallogenic province that hosts numerous world-class copper-cobalt deposits both in the DRC and Zambia Figure 7.2).

Figure 7.1 Regional Geological Setting of the Lufilian Arc and Location of the Kipushi Project in the Central African Copperbelt



Modified after Kampunzu et al., 2009

Figure 7.2 Structural Domains and Schematic Geology of the Central African Copperbelt, and the Location of the Kipushi Project



Ivanhoe Mines, 2015; adapted after François, 1974

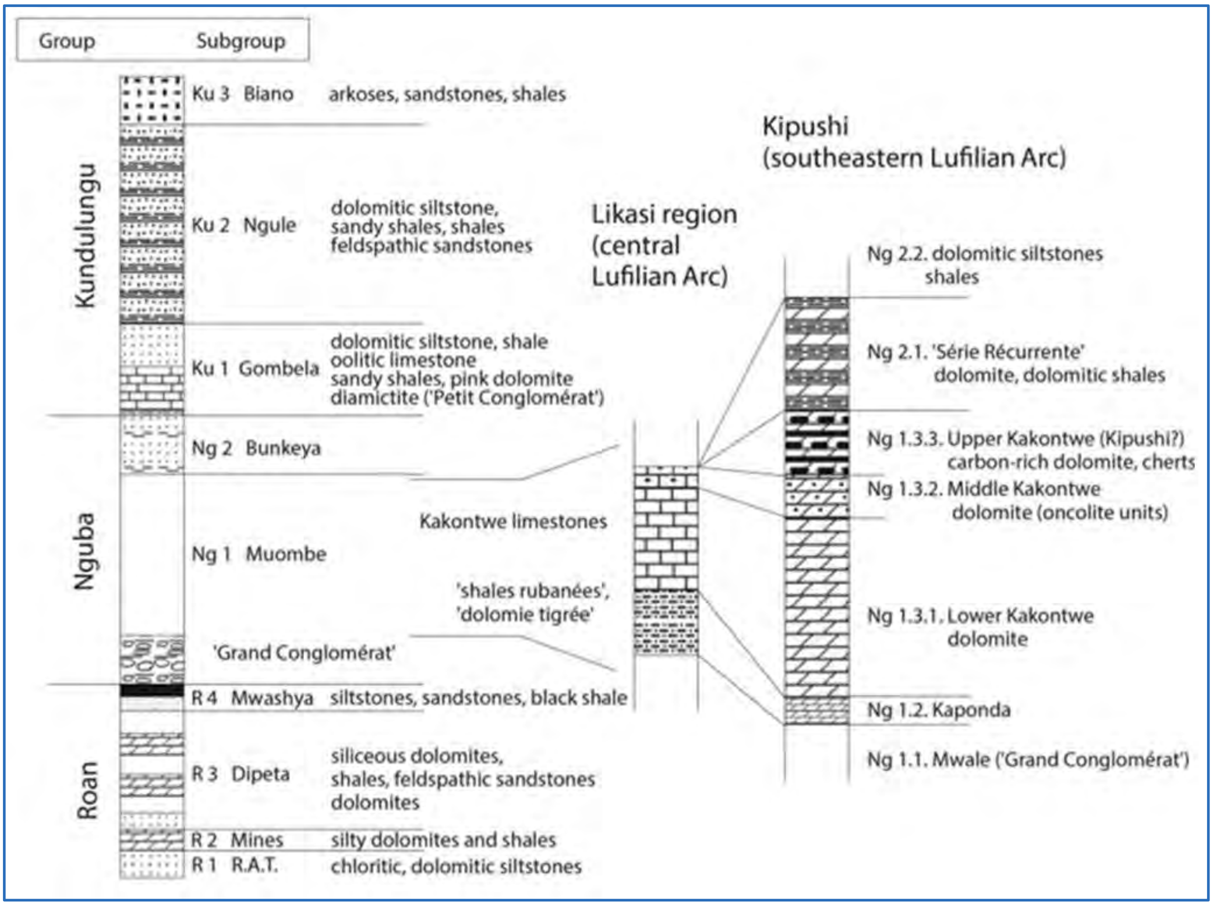
The Central African Copperbelt is contained in the Katangan basin, an intracratonic rift that records onset of growth at ~840 Ma and inversion at ~535 Ma (Selley et al., 2018). The lowermost sequences were deposited in a series of restricted rift basins that were then overlain by laterally extensive, organic rich, marine siltstones, and shales. This horizon is overlain by what became an extensive sequence of mixed carbonate and clastic rocks of the Upper Roan Group (Selley et al., 2005). The extensional geometry was preserved through orogenesis, forming what is known as the Lufilian Arc. The arc geometry, similar in character to oroclinal bending, has conventionally been interpreted to be composed of a stack of thin-skinned, north verging fold and thrust sheets (e.g. François and Cailteux, 1981; Kampunzu and Cailteux, 1999), however other work (De Magnee and François,1988; Jackson et al., 2003; Selley et al., 2018) favours a salt tectonic origin for the Copperbelt geometry. The crustal scale Mwembeshi Dislocation Zone separates the Lufilian Arc from the Zambezi Belt to the south.

The underlying basement comprises Neoarchaean granites and granulite's of the Congo Craton in the western part of the Lufulian Arc, and Palaeoproterozoic schists, granites and gneisses of the Domes Region, the Lufubu Metamorphic Complex, and the quartzite-metapelite sequence of the Muva Supergroup in Zambia (Kampunzu et al., 2009).

7.1.1 Stratigraphy

The Katanga Supergroup is subdivided into three major stratigraphic units: the basal Roan, the middle Nguba (formerly known as the Lower Kundulungu), and the uppermost Kundulungu Groups. These are separated on the basis of two regionally correlated (glaciogenic) diamictite units. The stratigraphy of the Katanga Supergroup as defined in the traditional DRC context, is shown in Figure 7.3.

Figure 7.3 Stratigraphy of the Katangan Supergroup, Southern DRC



Heijlen et al., 2008

The Roan Group was deposited unconformably on the basement. The youngest rocks include zircons in the basal sequence in Zambia and give a maximum 880 Ma age for sedimentation (Armstrong et al, 2005). The base of the Roan sequence in the Congolese Copperbelt is not exposed or drilled, and as identified consists of a lower siliciclastic unit (Roches Argilo-Talqueuses (R.A.T.) Subgroup) inferred to also have contained evaporites, a middle carbonate and siliciclastic unit (Mines Subgroup), an upper carbonate unit (Dipeta Subgroup), and an uppermost siliciclastic to calcareous unit (Mwashya Subgroup). Stratigraphic relations, particularly between these Subgroups, are commonly obscured by unusual breccias considered to be evaporitic in origin.

The Nguba Group comprises a lower siliciclastic and dolomitic limestone unit (Muombe Subgroup) and an upper predominantly siliciclastic and minor calcareous unit (Bunkeya Subgroup). The base of the Nguba Group is marked by a regionally extensive matrix-supported glaciogenic diamictite known as the Grand Conglomérat, referred to as the Mwale Formation. Zircons from sparse included peperites intruded into the basal un-lithified diamictite provide U-Pb ages of 735 Ma ± 5 Ma (Key et al., 2001). The overlying dolomitic limestones (Kaponda or Lower Kakontwe, Middle Kakontwe, and Kipushi or Upper Kakontwe Formations) are the hosts to Zn-Pb-(Cu) mineralisation in the DRC. The overlying Bunkeya Subgroup comprises the Katete (Série Récurrente) and Monwezi Formations, which are made up of dolomitic sandstones, siltstones, and shales.

The Kundulungu Group is subdivided into three subgroups in the DRC, comprising a lower siltstone-shale-carbonate unit (Gombela Subgroup), a middle dolomitic pelite-siltstone-sandstone unit (Ngule Subgroup), and an upper arenaceous unit (Biano Subgroup) interpreted as a molasse sequence. The base of the Gombela Subgroup is marked by a second regionally extensive matrix-supported glaciogenic diamictite (Petit Conglomérat) which is overlain by a dolomitic limestone cap. The diamictite is correlated to the global Marinoan glaciation dated by Hoffman et al., (2004) to 635 Ma from a recognised equivalent in Namibia.

7.1.2 Mineralisation and Tectonic Evolution

The largest Cu ± Co ores, both stratiform and vein-controlled, are known from the periphery of the basin and transition to U-Ni-Co and Pb-Zn–Cu ores toward the deepest portion of the basin. Most ore types are positioned within a ~500 m halo to former near-basin-wide salt sheets or associated salt movement (halokinetic) structures. Mineralisation in the majority of the Katangan Copperbelt orebodies such as at Kolwezi and Tenke–Fungurume (Figure 7.2) is hosted in the Mines Subgroup (R2). The mineralisation at Kipushi differs from these deposits in that it is located in the stratigraphically higher Nguba Group.

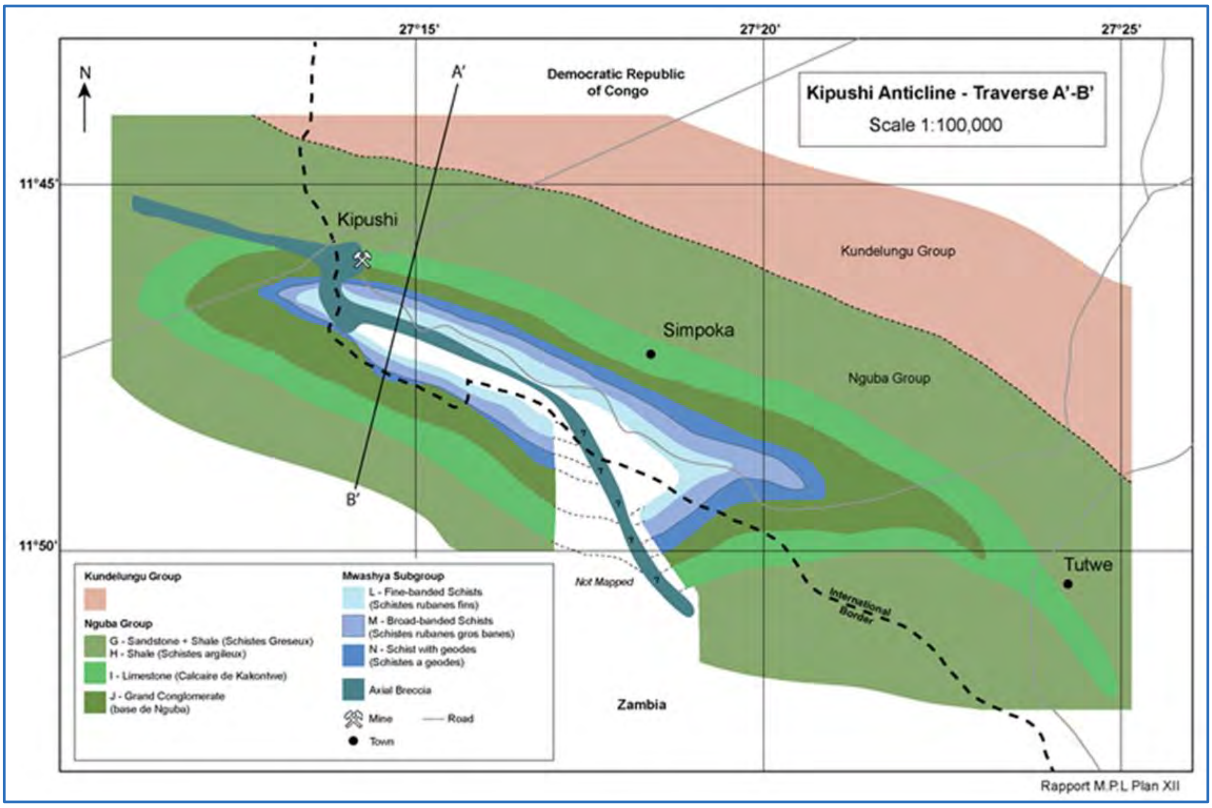
Mineralising fluids appear linked to residual evaporitic brines generated during deposition of the basin-wide salt-sheets, occupying large sub and intrasalt aquifers from ~800 Ma. This marks the earliest likely mineralising event, particularly in the Zambian-type stratiform Cu ± Co ores (Selley et al., 2018). At variable times from ~765 to 740 Ma, movement in the salt sheets in the Congolese part of the basin caused their modification allowing deeper level residual brines to interact with reducing elements and form the stratiform ores (Selley et al., 2018).

Vein and/or fracture-hosted mineralisation types (e.g. Tilwezembe and Kipushi) are widespread across the Congolese portions of the basin and are always associated with salt tectonic-related breccias (Selley et al., 2018). Unlike other Copperbelt deposits, Kipushi is considered to be the youngest deposit (~450 Ma based on a well-constrained Re-Os Zn–Cu-Ge age for at least one stage of mineralisation reported by Schneider et al., 2007). This post-dates orogenesis, yet the mineralising fluids still contain a strong halite dissolution signature (Heijlen et al., 2008).

7.1.3 Structure

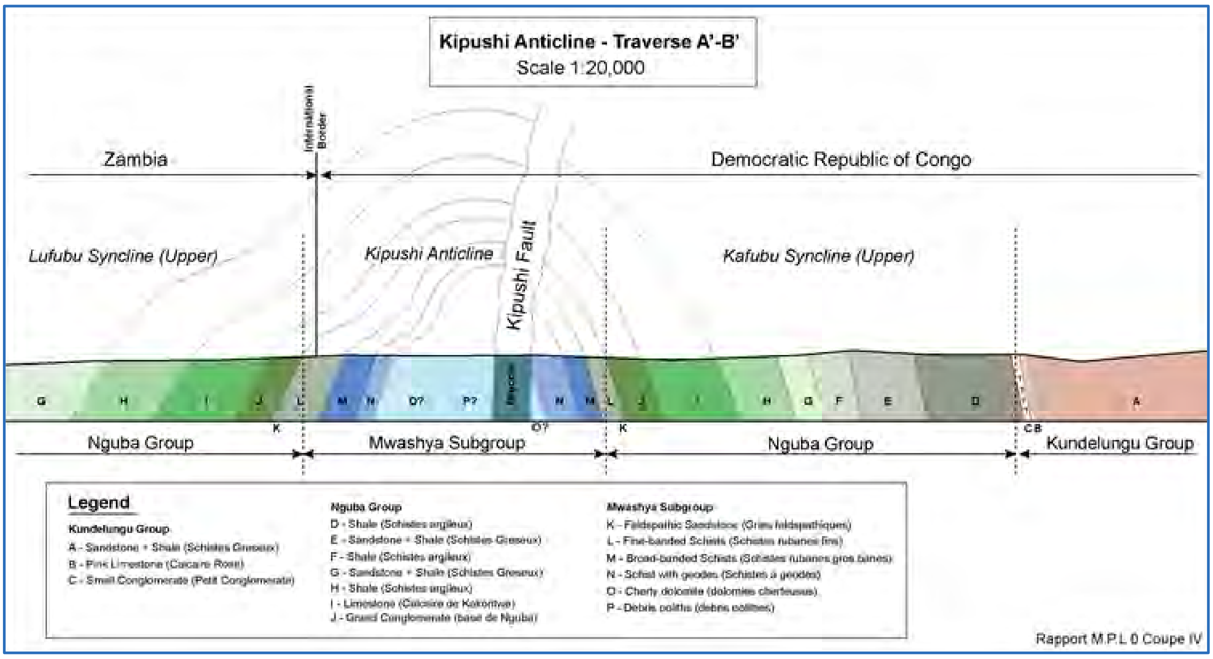
The Kipushi Project is located on the northern limb of the regional west–north-west trending Kipushi Anticline which straddles the border between Zambia and the DRC. The northern limb of the anticline dips at 75–85° to the north–north-east and the southern limb at 60–70° to the south–south-west as shown in the cross-section in Figure 7.4 and Figure 7.5. The anticline has a faulted axial core comprising a megabreccia referred to as the 'Axial Breccia' by Kampunzu et al., (2009). The megabreccia occurs as a heterogeneous layer-parallel breccia with highly strained and brecciated fragments of Roan and Nguba Group rocks in a chloritic silty matrix (Briart, 1947). This breccia type is similar to those typically associated with salt movement tectonics, and first proposed as such by de Magnee and François (1988).

Figure 7.4 Geological Map of the Kipushi Anticline



Ivanhoe Mines, 2015; adapted after Briart, 1947

Figure 7.5 Section Through the Kipushi Anticline



Ivanhoe Mines, 2015; adapted after Briart, 1947

The northern limb of the Kipushi anticline dips approximately 80° north, considerably steeper than the southern limb. The steeply southern dip of the anticline axial plane is paralleled by a slatey cleavage, well developed in the siltstones of the Katete formation, and expressed as an anastomosing spaced cleavage in the Upper Kakontwe formation both believed to have developed during north–north-east directed compression Figure 7.6.

Bedding Dips Steeply to north–north-west (here in proximity to Kipushi Fault) and is cut by a Steep East–West Cleavage. Core is positioned such that the image represents a Plan View with North to the top (Figure 7.6).

Figure 7.6 Interbedded Dolomite-Shale/Siltstone Unit in the Upper Kakontwe Formation at 153 m in KPU070 (hole orientation –35 to 125)

Ivanhoe Mines, 2015

7.2 Local Geology

There is abundant literature focussing on mineralogy and geochemistry at Kipushi (e.g. Heijlen et al., 2008; Kampunzu et al., 2009, and references therein), but a paucity of modern work and literature relating to stratigraphy, structure, and interpretation of the host rocks. Intiomale (1982) and Intiomale and Oosterbosch (1974) have served as the primary references for the stratigraphic and geological description of the deposit. These in turn heavily reference a report by Union Minière du Haut Katanga published in 1947 (Briart, 1947) and held in Teuveren, Belgium. Much of this work predates or ignores ideas of allochthonous salt that were introduced in the Copperbelt in the late 1980s (De Magnée and François, 1988), and more recent work (Selley et al., 2005) relating to the importance of growth-faults in basin evolution. Work by Turner et al. (2018) has begun to address this lack of modern literature, with an update on the geological understanding of the depositional environment for the Kipushi deposit.

The only surviving production-era geological maps at the Kipushi Mine are level plans, on which structural data are few, mainly recording strike and dip, and the upper contact of the Kakontwe Formation. Systematic underground mapping, if conducted, is no longer preserved, and surviving level plans and drill sections were historically interpreted primarily on the basis of interpolation between drillholes. Therefore, the geological model has been developed from the current drill programme and reinterpretation of existing historical data, including drill cores.

7.2.1 Stratigraphy

The stratigraphic sequence at Kipushi forms part of the Nguba Group, whose maximum depositional age is constrained by zircons from mafic rocks intruded into the basal unlithified diamictite providing U-Pb ages of 735 Ma ± 5 Ma (Key et al., 2001). This is succeeded by a carbonate-dominant sequence of the Kaponda and Kakontwe Formations that attain a thickness of greater than 600 m at Kipushi, considerably greater than elsewhere in the Congolese Copperbelt. The overlying Katete Formation (Série Récurrente) consists of alternating greenish siltstone and pale purple dolostone.

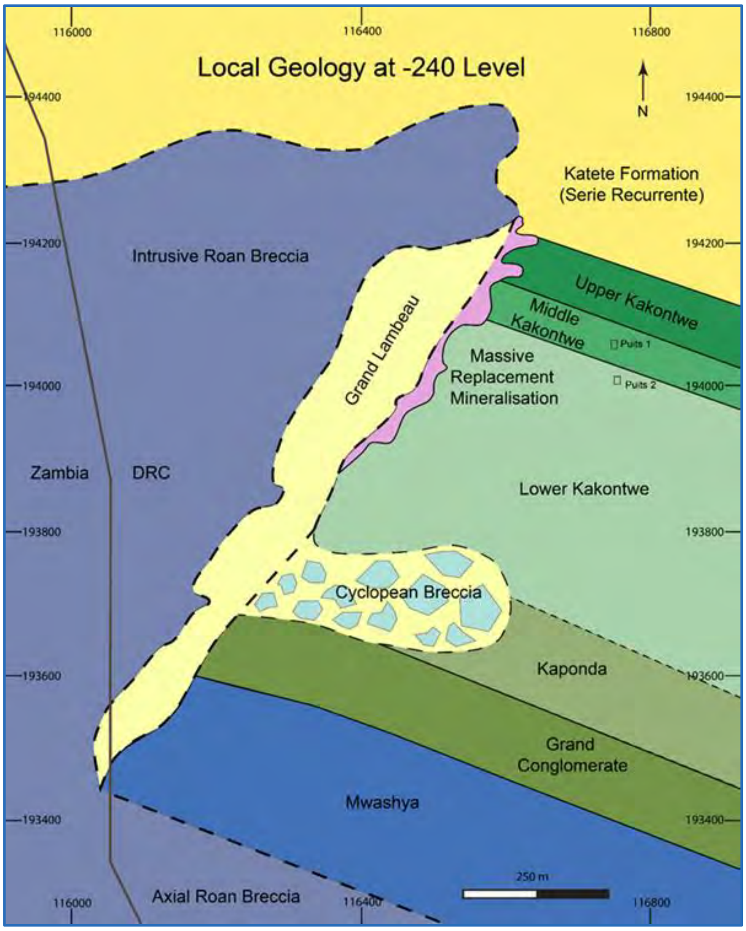
The Lower Kakontwe (LK) is a massive pale grey unit that consists almost entirely of microbial limestone. A range of variably preserved muddy-laminated (irregularly undulatory) to very distinctive calcimicrobial (laminated and clotted) lithofacies is present, with lesser volumes of featureless carbonate mudstone and rare microbialite-derived intraclasts. The Middle Kakontwe (MK) is a monotonous, dark grey, fine grained carbonate mudstone lacking conspicuous bedding or sedimentary structures, while the Upper Kakontwe (UK) unit consists of dark grey, stratified carbonaceous carbonate mudstone. The lower part of the Upper Kakontwe Formation consists of interlayered thin-bedded to laminated carbonate mudstone, carbonaceous black shale seams, and rudstone-floatstone layers of angular carbonate clasts up to 100 cm in diameter (Brooks, 2015).

KICO's drilling has only intersected the base of the Katete or Série Récurrente Formation. It is comprised of alternating beds of distinctive greenish-grey shale and purplish dolomite, of approximately a metre thickness (Brooks, 2015).

The Grand Lambeau ('large fragment') (GLB) unit comprises medium green/grey, interbedded siltstone and sandstone that is occasionally calcareous or dolomitic. A variety of sedimentary textures are present, including graded beds, brittle dewatering structures (cracks), syn-sedimentary micro faults, and soft-sediment deformation (Brooks, 2015). The bedding in the GLB generally parallels that of the Kakontwe Formation bedding to the east, dipping steeply to the north–north-east at ~70°.

The relationships of these units are shown in the schematic representation of the Kipushi deposit at the –240 mL in Figure 7.7. The Katangan sequence has been rotated during the formation of the Kipushi anticline, therefore, the plan view shown in Figure 7.7 is analogous to a pre-folding approximately north-west–south-east section view. This configuration changes remarkably little in section, down to at least 1,200 m depth.

Figure 7.7 Schematic Geological Map of the Kipushi Deposit at a Depth of 240 m below Surface



Ivanhoe Mines, 2018; adapted after Briart, 1947

The carbonaceous breccia and fault zone siltstone-shale are believed to represent Upper Kakontwe strata entrained within the fault zone that has undergone subsequent dissolution of the carbonate during reactivation, leaving only clay and organic carbon (Figure 7.13).

7.2.2 Kipushi Reef Edge Interpretation

Various authors (Briart, 1947, Intiomale, 1982, Kampunzu et al., 2009) have described the so-called 'Kipushi Fault Zone'. They viewed it as a 10–50 m wide complex structure recording multiple styles of deformation and brecciation, separating the footwall Kakontwe Formation from the hanging wall Grand Lambeau, which is described as a km-scale block of stratified carbonate-rich shales, siltstones, and fine-grained sandstones of the Kiubo Formation (Kundulungu Group) enclosed in the 'Cyclopean Breccia' (Figure 7.7). KICO inherited this interpretation and originally envisaged the Kipushi Fault as a complex, multistage zone predicated on a syn-sedimentary growth fault that was reactivated during subsequent tectonic events, such as the development of the Kipushi anticline.

The Kipushi orebodies are located along this, approximately north–north-east striking, west dipping (~70°), brecciated, fault like feature (Figure 7.11). It has an irregular, highly sinuous geometry, such that the location and orientation of its hanging wall and footwall contacts vary, commonly independently, along strike and down dip. The siltstones and sandstones of the Grand Lambeau are truncated on their western side by the intrusive axial breccia (Figure 7.7).

The KICO drilling campaigns of 2014 and 2017 added a large amount of new drill core to the historic La Générale des Carrières et des Mines (Gécamines) drilling. Using all this latest information, Turner et al., (2018) reviewed the 'Kipushi Fault' and provided an alternative interpretation that has been adopted by KICO.

Turner et al., (2018) suggested that the abundant microbial textured dolomite of the Lower Kakontwe represents late stage reef growth that established a depositional escarpment at the reef edge. This resulted in considerable relief above the contemporaneous sea floor where unrelated deep-water sediment accumulated.

The Lower Kakontwe formed the carbonaceous reef, with the Middle and Upper Kakontwe formations forming the non-reefal cap carbonates, with a gradational transition into the overlying non-carbonaceous Série Récurrente. Turner et al. (2018) asserts that the siliciclastic sequence of the Grand Lambeau formed contemporaneously to the Kakontwe Formations off the reef edge and is not a stratigraphically out of sequence fragment from higher up in the Katanga Supergroup, as described by previous authors (Briart, 1947, Intiomale, 1982, Kampunzu et al., 2009).

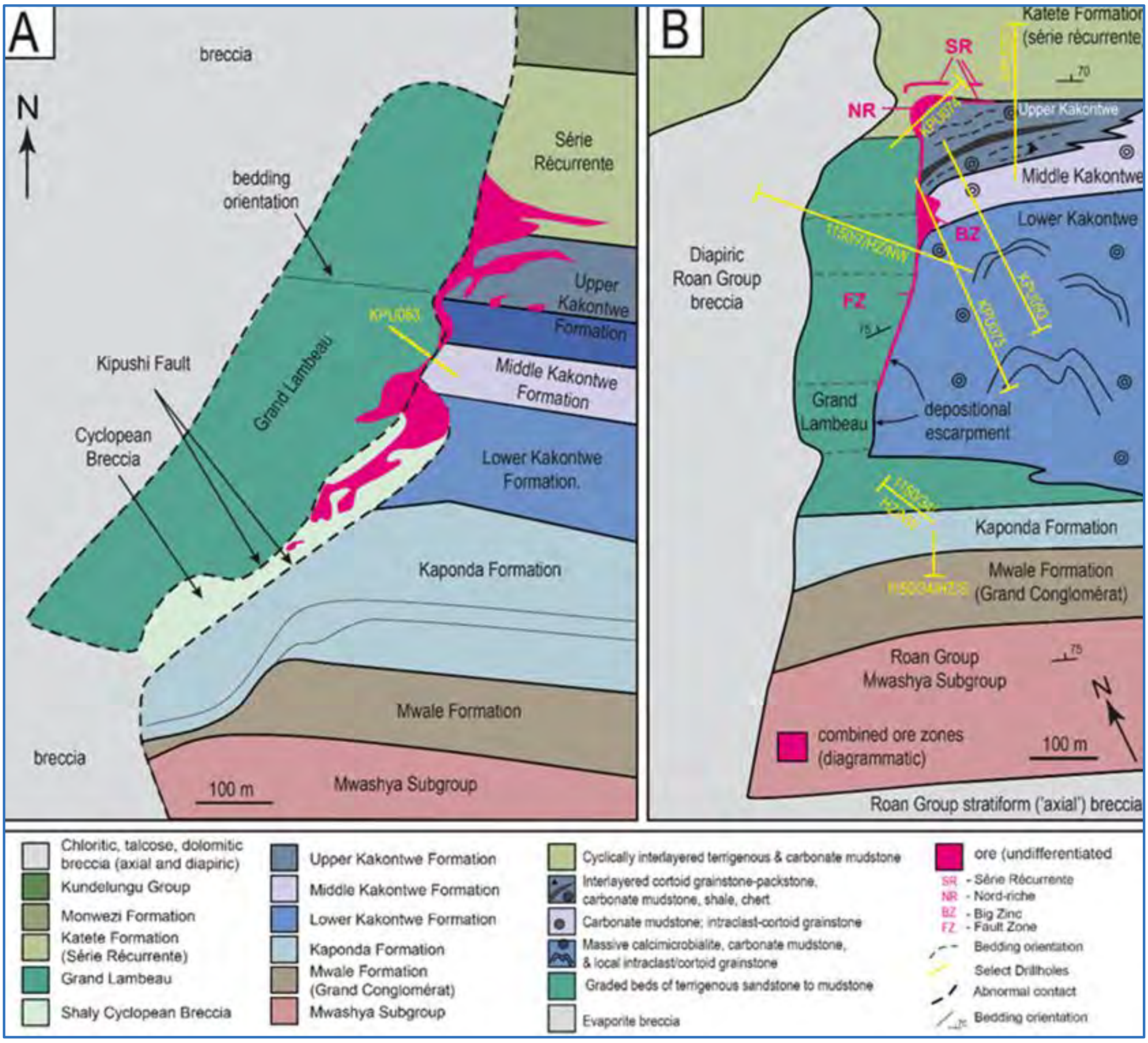
Various factors provide evidence to support this idea: (1) the fact that the Grand Lambeau and the strata overlying the Série Récurrente are identical (Turner et al., 2018), (2) The bedding in the Kakontwe dolomites and the Grand Lambeau siltstone and sandstone units are roughly coplanar (Briart, 1947, Intiomale, 1982, Kampunzu et al., 2009, Turner et al., 2018 and KICO geologists), (3) The clasts within the brecciated lithology along the Grand Lambeau contact are made up of clasts from the corresponding adjacent Kakontwe units (either the Lower, Middle or Upper) or from the Série Récurrente, depending where along the reef edge you are.

Turner et al., 2018 contend that these factors argue for a "depositional origin for the Kipushi carbonate escarpment and the penecontemporaneous deposition of the Grand Lambeau against the flank of the growing carbonate bank, and depositional draping to the Série Récurrente atop both of these underlying units" (Figure 7.8).

The Fault Zone is characterised by a breccia, which could have accumulated as clasts, or blocks, of the lithified reef which slumped or fell down the reef edifice to accumulate within a silty matrix at the base of the reef escarpment. Augmenting this, the juxtaposition of the carbonaceous Kakontwe and siliciclastic Grand Lambeau is marked by a large, permanent rheological contrast, and persistent zone of structural weakness (Turner et al., 2018). As a result, this could have led to further brecciation of the contact zone over multiple reactivations.

Based on the above description, as well as core and underground observations, this variable bedded and brecciated zone does not appear to be a fault, in the strict sense. The Fault Zone nomenclature, however, has been used for nearly 100 years since the development of the mine and will remain in use by KICO.

Figure 7.8 Level Plan Showing Historical Interpretation



Ivanhoe Mines, 2018; adapted after Turner et al, 2018
Compared to the Updated Reef Edge Interpretation (B) for the Kipushi Fault Zone (FZ).

A description of the Kipushi stratigraphy and traditional alpha-numeric nomenclature is given in Table 7.1, with this coding method maintained by KICO during geological logging.

Table 7.1 Updated Stratigraphic Column for the Kipushi Project

Reef Stratigraphy						
Subgroup		Formation		Description	Traditional Congolese Designation	Mineralisation
Upper Nguba (Bunkeya)	Monwezi	Katete Formation (Série Récurrente		Laminated, purple to whitish, albite-bearing calcareous and talcose dolostone with interbedded grey-green to dark grey shale bands.	Ki2.1	Layer parallel, concordant disseminated and blebby cpy with minor bnt, typically <2% Cu with minor Mo and Re
Lower Nguba (Muombe)	Kipushi	Termed Upper Kakontwe by KICO and GCM	Kipushi Formation	Finely bedded black carbonaceous dolomite unit, up to 100 m thick (e.g. at Kipushi), characterised by blark chert lenses and whitish oncolites, slump structures and lenticular grey-brown dolomitic shale. ~50 m thickness.	Ki1.2.2.3 (Ki1.2.2.4)	Discordant massive and replacement cpy and minor sph.
	Kakontwe		Upper Kakontwe	Kakontwe unit is a dark grey, stratified, calcareous and carbonaceous dolostone with intercalations of fine carbonaceous layers and black cherts. ~50 m thickness (thickens with depth)	Ki1.2.2.3	Disordant massive and replacement cpy and minor sph
		Middle Kakontwe		Massive and occasionally finely bedded carbonate mudstone. Oncolites at upper contact. ~80 m thick.	Ki1.2.2.2	Discordant massive and replacement sph with minor cpy
		Lower Kakontwe		Light grey massive lamelliform and dotted calcimicrobial carbonates with a variety of textures. ~250 m thick.	Ki1.2.2.1	Discordant massive and replacement sph with minor cpy

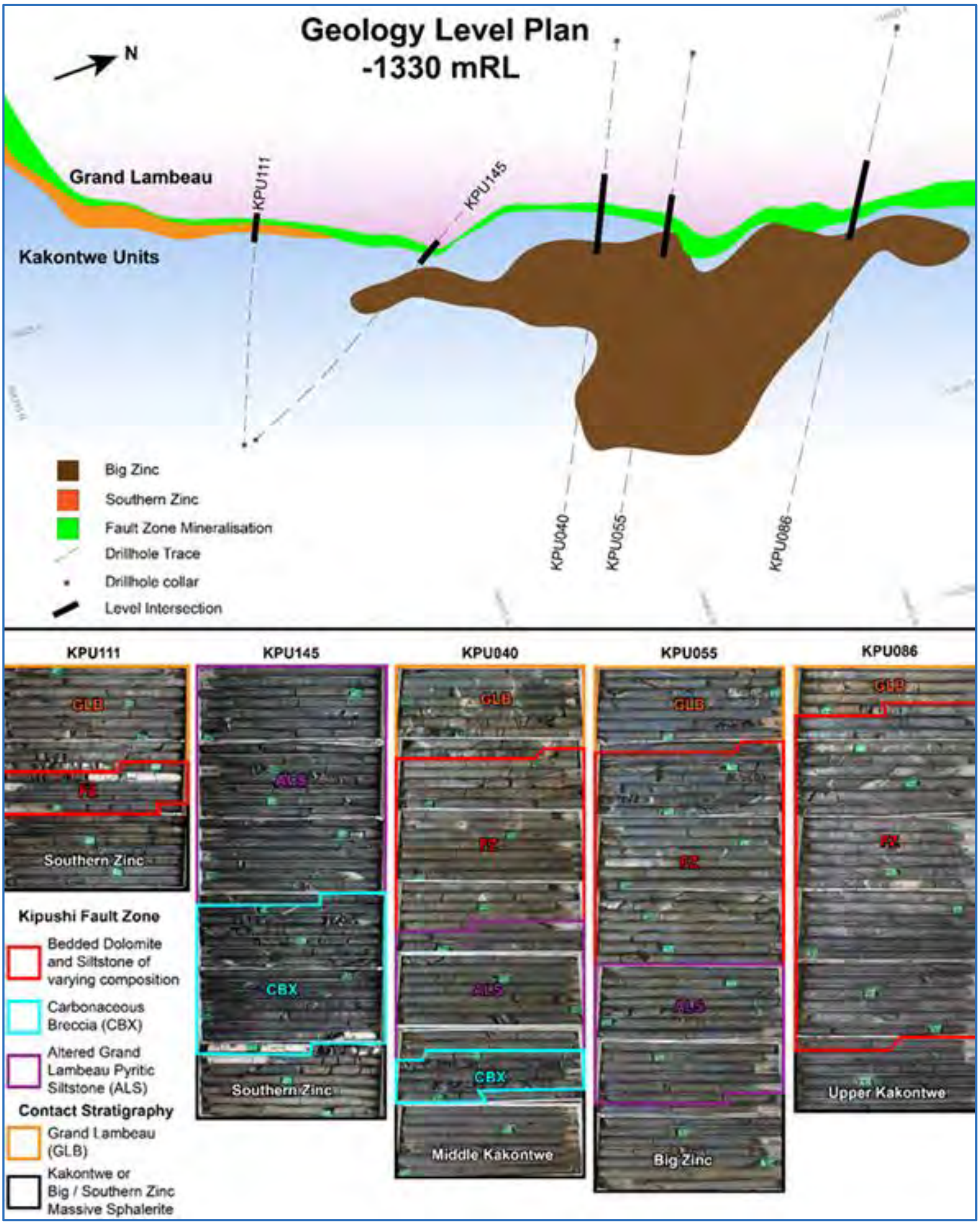
Reef Stratigraphy						
Subgroup		Formation		Description	Traditional Congolese Designation	Mineralisation
	Kaponda	Kaponda Formation		Finely laminated blue-grey to dark grey, sometimes cherty, and carbonaceous dolostone, calcareous in places. Dark, tortuous, lenticular cherty and dolomicritic layers alternating with lighters dolomicritic layers forming 'Dolomite de Tigre' (Tiger Dolomite) pattern.	Ki1.2.1	
Off Shelf, Deep Stratigraphy						
Upper Nguba (Bunkeya)	Monwezi	Off Reef Facies	Grand Lambeau	Fine grained sandstones, siltstones, and minor calc-arenites of the 'Grand Lambeau'		Minor cpy and sph associated with the Breccia zone, and the Kakontee contact.

The Fault Zone (FZ) nature (thickness, lithological, mineralisation composition, and style) changes from north to south across the deposit. The feature has a sinuous morphology (Figure 7.9) along both strike and dip. It generally dips at ~70° to the west, but locally it can be vertical or even dip slightly to the east. The thickness of this zone generally decreases from north to south with the thickest occurrence (~50 m) in the north, near the Upper Kakontwe – Série Récurrente contact, thinning to ~1 m in the Lower Kakontwe.

There is large lithological variability across the zone, ranging from a thin (metre scale) carbonaceous breccia (CBX) in the south, to thick (tens of metres) interbedded dolomite and siltstone in the north (Figure 7.9). The drillholes shown in Figure 7.9 are representative of the type of lithological variation seen within the FZ. The figure shows the general range of lithologies observed across the deposit. The CBX is predominantly found in the southern portion of the deposit, where it typically constitutes the entire zone (KPU145), with minor variations of bedded carbonaceous dolomite observed (KPU111). The CBX is seen to decrease to the north, with KPU040, containing both the CBX and interbedded dolomite and shale. In the north, the zone thickens up and becomes purely interbedded dolomite and siltstone (e.g. KPU055) and in the far north, close to the Série Récurrente contact, purple dolomite becomes dominant. It is clearly seen that the dolomitic composition differs across the extent of the zone. The Grand Lambeau is commonly seen to be altered on the FZ contact, and can contain significant pyrite (KPU040, KPU055, and KPU145).

Within the CBX, both the matrix and clasts present in the breccia are variable, depending where you are in relation to the reef stratigraphy. There is also a contrast of the clast size and roundness within the breccia. The style and intensity of deformation is variable, both within the same intersection and in neighbouring intersections. The spectrum of brecciation is shown in Figure 7.10. The structural features observed also indicate that the deformation postdates the brecciation. The Fault Zone has long been recognised as a locus for mineralisation, which is predominantly copper rich, with minor zinc mineralisation, and in most cases relatively pyritic.

Figure 7.9 Lithological Variation Across the Kipushi Fault Zone at the –1,330 mL



Ivanhoe, 2018; Note the intersections shown in the core photographs in the bottom half of the image are the drilled intersection and not true thickness.

Figure 7.10 Structural and Mineralisation Variability Within the Kipushi Fault Zone

Ivanhoe Mines, 2018; Carbonaceous Breccia variations from KPU115.
(A) Un-deformed breccia with carbonaceous matrix and angular dolomite clasts, (B) Breccia with aligned dolomite clasts and minor boudinage structures, (C) Sulfide replacement around dolomite clasts, with occasional mineralisation shadows.

7.3 Alteration and Metamorphism

The rocks at Kipushi appear to have experienced lower greenschist facies metamorphism.

Kipushi has a unique alteration signature for the Copperbelt, with a multistage assemblage of dolomite, quartz, Ba feldspar, Ba muscovite, Mg chlorite, phlogopite, and muscovite (Chabu and Boulegue, 1992; Heijlen et al., 2008).

From the two drill programmes to date, alteration associated with mineralisation is observed to include 88rebody88zation of the Kakontwe Formation limestone up to 200 m away from the Fault Zone, silicification of wall rock dolomite, formation of black amorphous organic matter in the Kakontwe dolomite up to 40 m away, 88rebody88zation along mineralisation contacts and along fractures, and 88rebody88zatio of feldspars within the Grand Lambeau.

The Grand Lambeau that is in direct contact with the Fault Zone has experienced minor alteration, due to fluid flow along the contact. The alteration exhibits as a colour change from the typical beige sandstone / siltstone colouration to dark grey. This gradually dissipates within tens of metres from the contact.

7.4 Mineralisation

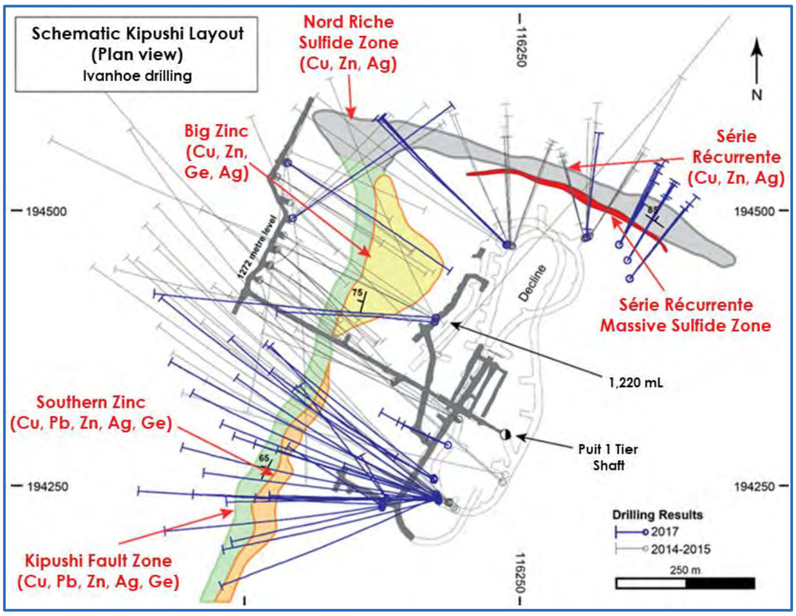
7.4.1 Overview

The Katanga Supergroup hosts a number of epigenetic zinc–copper-lead deposits developed within deformed platform carbonate sequences. While many of these are relatively small (e.g. Kengere and Lombe in the DRC; Bob Zinc, Lukusashi, Millberg, Mufukushi, Sebembere, and Star Zinc in Zambia), Kipushi and Kabwe in the DRC and Zambia respectively represent world class deposits with predominantly massive sulfide mineralisation contained within dolomitic limestone (Kampunzu, et al., 2009). These deposits are polymetallic with a typical Zn-Pb-Cu–Ag-Cd-V association and contain variable concentrations of As, Co, Mo, Rh, Ge, and Ga.

Mineralisation at Kipushi is spatially associated with the intersection of Nguba Group stratigraphy with the Kipushi Fault and occurs in several distinct settings (Figure 7.11):

- Kipushi Fault Zone (copper, zinc, and mixed copper–zinc mineralisation both as massive sulfides and as veins).
- Série Récurrente:
  - Disseminated to veinlet-style copper sulfide mineralisation).
  - A high-grade pod (massive copper and zinc sulfides).
- Copper Nord Riche (mainly copper but also mixed copper-zinc sulfide mineralisation, both massive and vein-style).
- Big Zinc (massive zinc sulfide with local copper sulfide mineralisation).
- Southern Zinc (poly-metallic massive sulfide).

Figure 7.11 Schematic Layout with the Location of Distinct Mineralised Zones at Kipushi

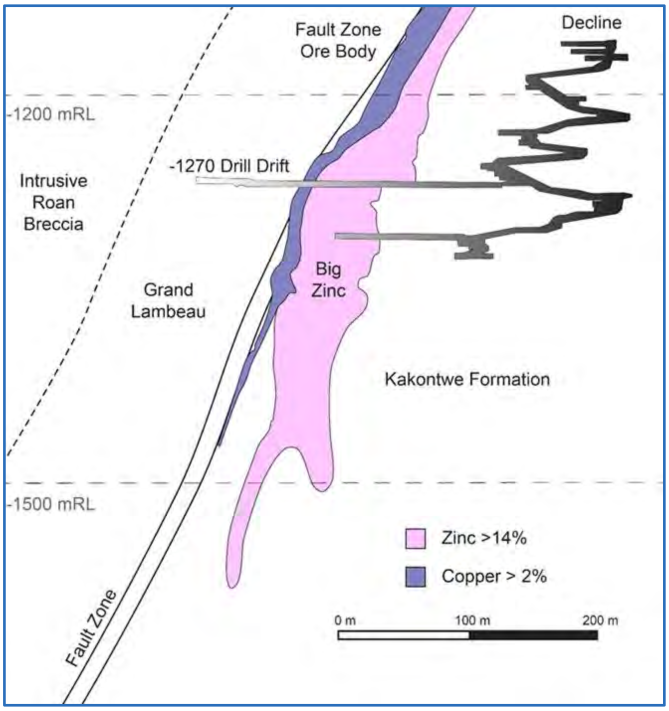


Ivanhoe Mines, 2018

Mineralisation at the Kipushi Project is generally copper-dominant or zinc-dominant with minor areas of mixed copper–zinc mineralisation. Pyrite is present in some peripheral zones and forms massive lenses, particularly in the Fault Zone. Copper-dominant mineralisation in the form of chalcopyrite, bornite, and tennantite is characteristically associated with dolomitic shales both within the Fault Zone and extending eastwards along, and parallel to, bedding planes within the Série Récurrente and adjacent Upper Kakontwe Formations.

Zinc-dominant mineralisation in the Kakontwe Formations occurs as massive, irregular, discordant pipe-like bodies completely replacing the dolomite host. These bodies exhibit a steep southerly plunge from the Fault Zone and Série Récurrente contacts where they begin, to their terminations at depth within the Kakontwe Formation (Figure 7.12). This southerly orientation, observed across all the mineralised zones, is oblique to the north-west plunging intersection of the Kakontwe Formations with the Fault Zone, inferring a persistent structural control at the Kipushi deposit.

Figure 7.12 Cross-Section Perpendicular to the Kipushi Fault, Looking North–North-East



Ivanhoe Mines, 2018

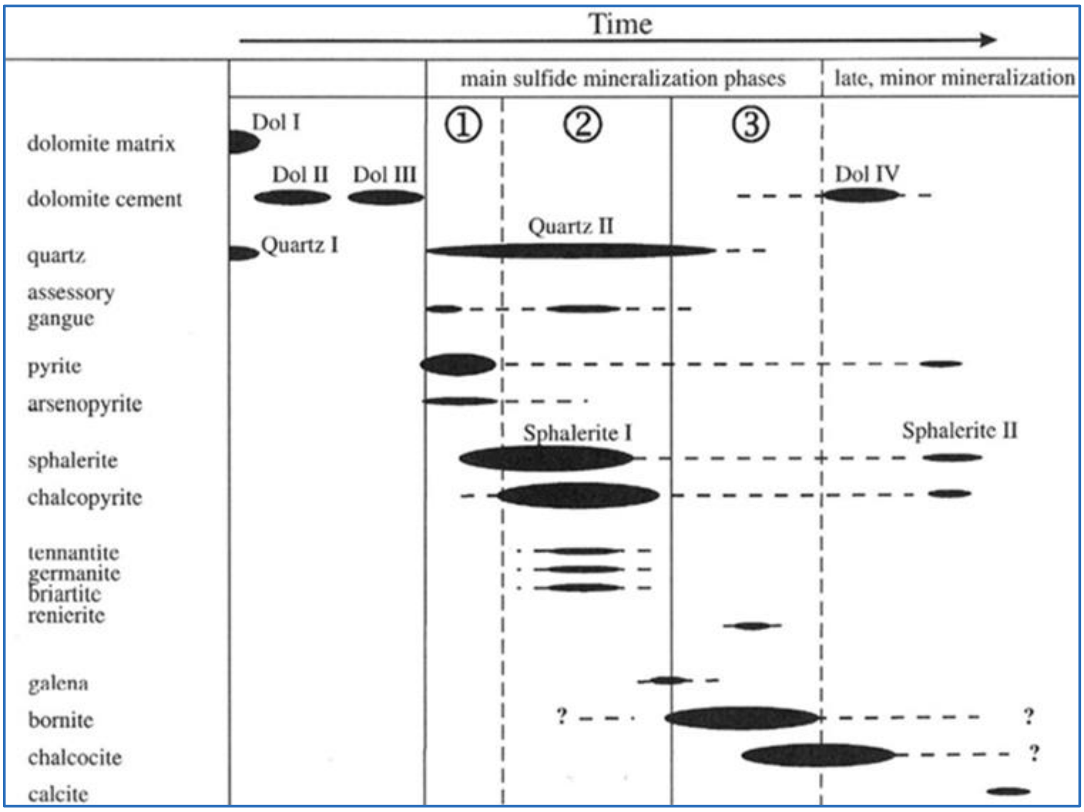
There is considerable variability in the mineralised zones, with a diverse range of economically significant accessory minerals for which Kipushi is well known. The complex mineralogy of Kipushi has been documented for over 60 years, although the lower levels of the deposit considered in this Kipushi 2019 Resource Update show simpler mineralogy.

Previous studies on the Kipushi mineralisation have shown that the sulfide mineralisation is complex and multiphase (e.g. Heijlen et al., 2008) and different generations of hydrothermal dolomite are also observed. A generalised paragenesis based on previous studies including work by Heijlen et al., (2008) is shown in Figure 7.13. As a typical feature, mineralisation formed through massive replacement of the dolomite host rock and cements, commonly resulting in banded mineralisation. Open space filling also occurred, but to a relatively minor extent. An initial sulfide phase of pyrite-arsenopyrite mineralisation was followed by sphalerite, chalcopyrite, tennantite, germanite, briartite, and galena in a second major phase of sulfide deposition. As a third major phase, bornite and chalcocite appear to selectively replace chalcopyrite as secondary mineralisation in the higher levels of the mine.

There is a clear sulfide zonation from copper-rich at the Fault Zone contact, to zinc-rich, to zinc and pyrite-rich massive sulfide at the contact with the Kakontwe Formation (left to right in Figure 7.14). This mineral zonation is similar to that seen in other Central African Copperbelt deposits, wherein copper is proximal to source (for example, the FZ) whereas zinc and pyrite are distal. Lead appears to be controlled, at least partially, by the Kakontwe stratigraphy, with the highest lead grades corresponding with the Upper–Middle Kakontwe contact.

The host dolomite has undergone extensive recrystallisation proximal to the mineralised zones and an increase in the silica content, with secondary grains and aggregates of fine quartz crystals (Chabu, 2003).

Figure 7.13 Generalised Paragenesis of Mineralisation and Gangue at Kipushi

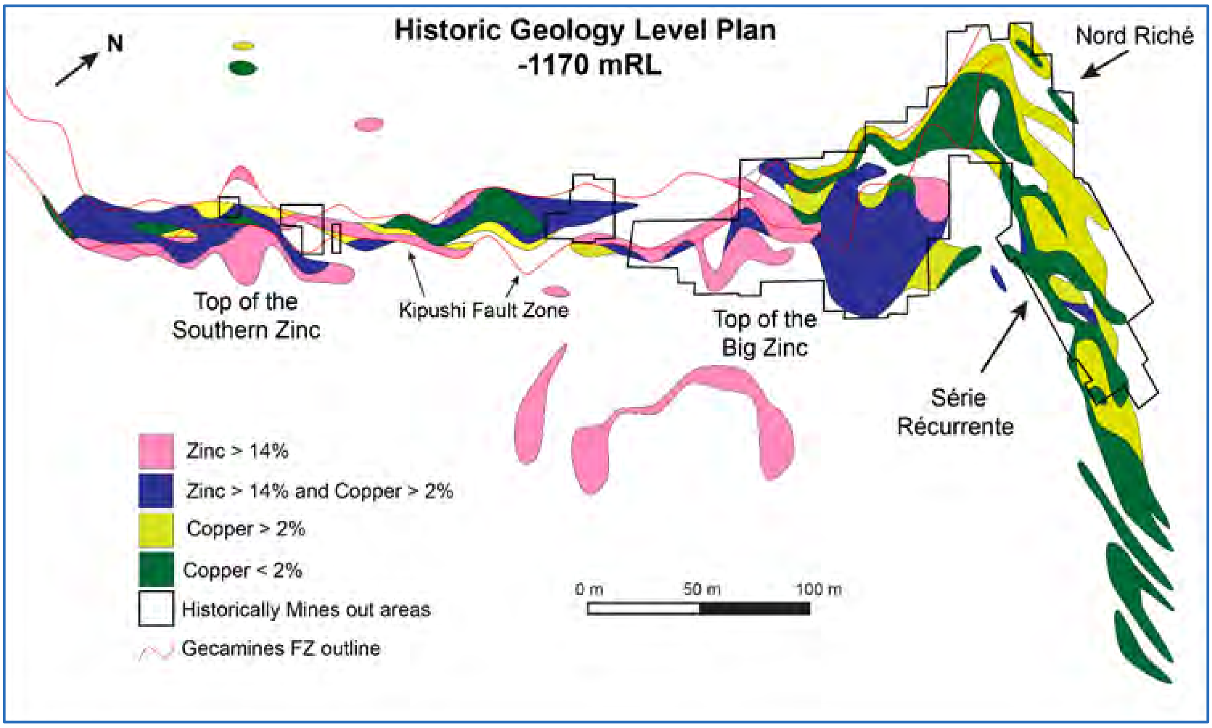


Heijlen et al., 2008

Historical mining at Kipushi was carried out from surface to approximately 1,220 mL and occurred in three contiguous zones, shown in Figure 7.14:

- Nord Riche area: The intersection of the roughly north–south trending Fault Zone and the approximately east–west striking Série Récurrente – Upper Kakontwe contact.
- The Fault Zone: south of the Nord Riche.
- The Série Récurrente (roughly east–west striking, steeply north dipping mineralisation): marking the contact between the Upper Kakontwe and Série Récurrente stratigraphic sequences.

Figure 7.14 Historic Level Plan from –1,170 mL, Showing the Different Historic Mining Zones



Ivanhoe Mines, 2018

7.4.2 The Big Zinc

The Big Zinc is a zone of massive sphalerite mineralisation in the Kakontwe Formations, best developed in the Middle Kakontwe (Figure 7.16). It is located in the immediate footwall to the Fault Zone between 1,100–1,700 mL. Mineralisation is discordant and occurs at least 100 m laterally along the footwall of the Fault Zone and extends up to 95 m into the footwall, near the Middle and Upper Kakontwe Formations' contact. The margins of the zone are characterised by a number of downward diverging 'apophyses' exhibiting a similar plunge to the rest of the Big Zinc (Figure 7.12). The zone diverges from the Fault Zone with increasing depth.

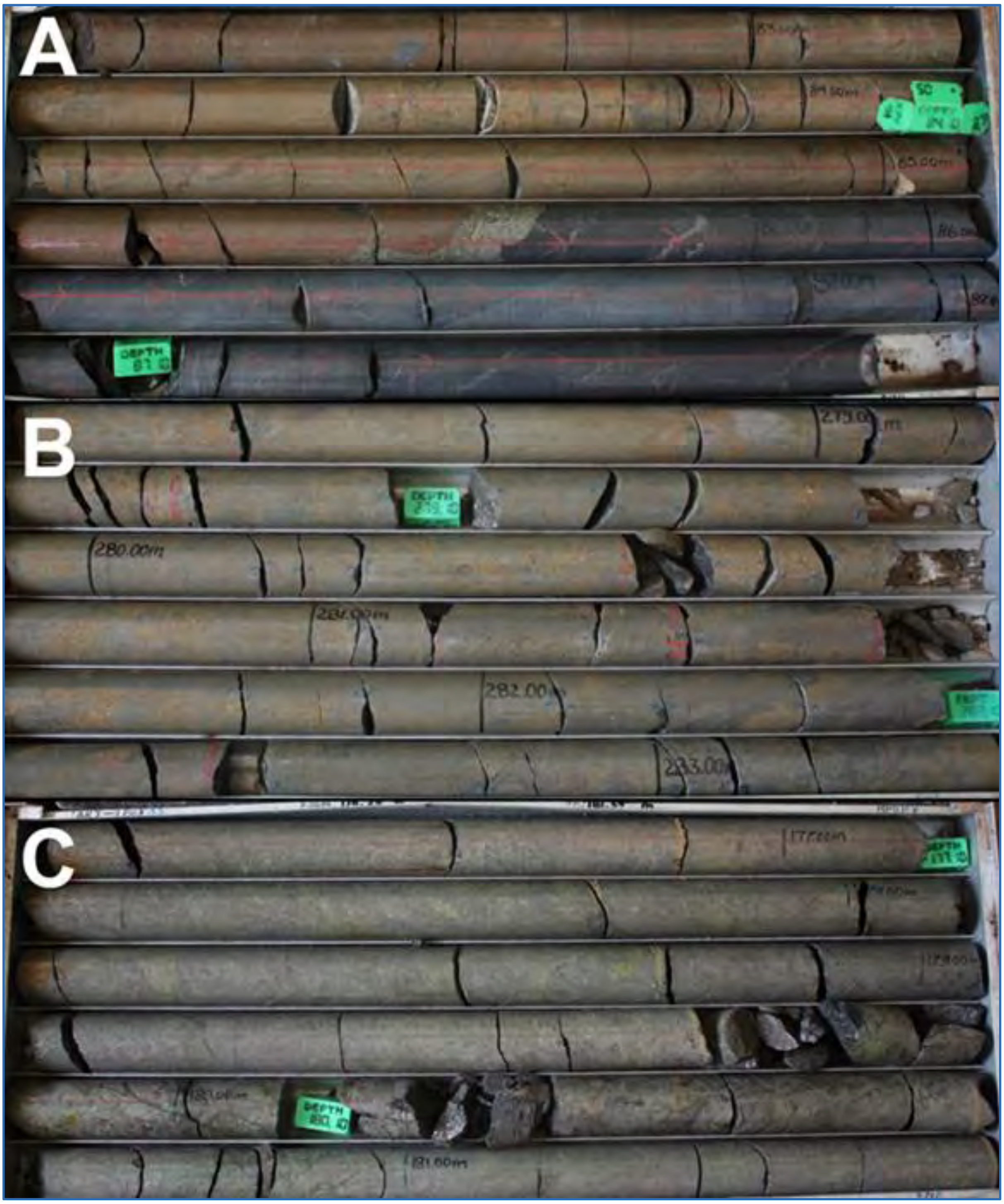
The contacts of mineralisation with the host Kakontwe dolomite are zoned over several metres. Sphalerite on the margins of the mineralised zone, particularly at the terminations of apophyses, is often red and iron-rich (Figure 7.15) and associated with arsenopyrite, and commonly grades outwards to a thin (centimetres to decimetres) outermost pyrite zone. Minor chalcopyrite and galena may also occur adjacent to the eastern and down-plunge margins. The outer (distal to the Fault Zone) contacts are occasionally marked by an abundance of distinctive megacrystic and mosaic-textured white hydrothermal dolomite inter-grown with the sulfides.

The Big Zinc is mineralogically simple with the majority of the deposit comprising massive, monotonous, equigranular to occasionally banded, honey-brown sphalerite and pyrite (Figure 7.15). Mineralisation textures commonly do not reflect primary textures within the host in any way. The sphalerite is zinc-rich (>45% Zn), iron-poor, and contains minor amounts of cadmium, silver, germanium, and mercury. The majority of the Big Zinc is hosted within the Middle Kakontwe Formation (Figure 7.16). The northern portion of the deposit is in the Upper Kakontwe Formation, and hosts disseminated galena and tends to be more silver-rich than the southern side. Germanium enrichment is irregular, but more common on the southern side of the Big Zinc and at depth, particularly in very zinc-rich sphalerite (Figure 7.17). Very high-grade (>55% Zn) and germanium rich (>100 ppm Ge) sphalerite is not visually distinguishable from the majority of sphalerite within the Big Zinc.

Tennantite, bornite, and chalcopyrite locally replace sphalerite in a 10–20 m thick pod of >100 m plunge extent within the Big Zinc (Figure 7.17). Smaller zones of tennantite mineralisation occur elsewhere in the Big Zinc, Copper Nord Riche, and Série Récurrente. These zones are associated with very high silver, cobalt, and molybdenum grades.

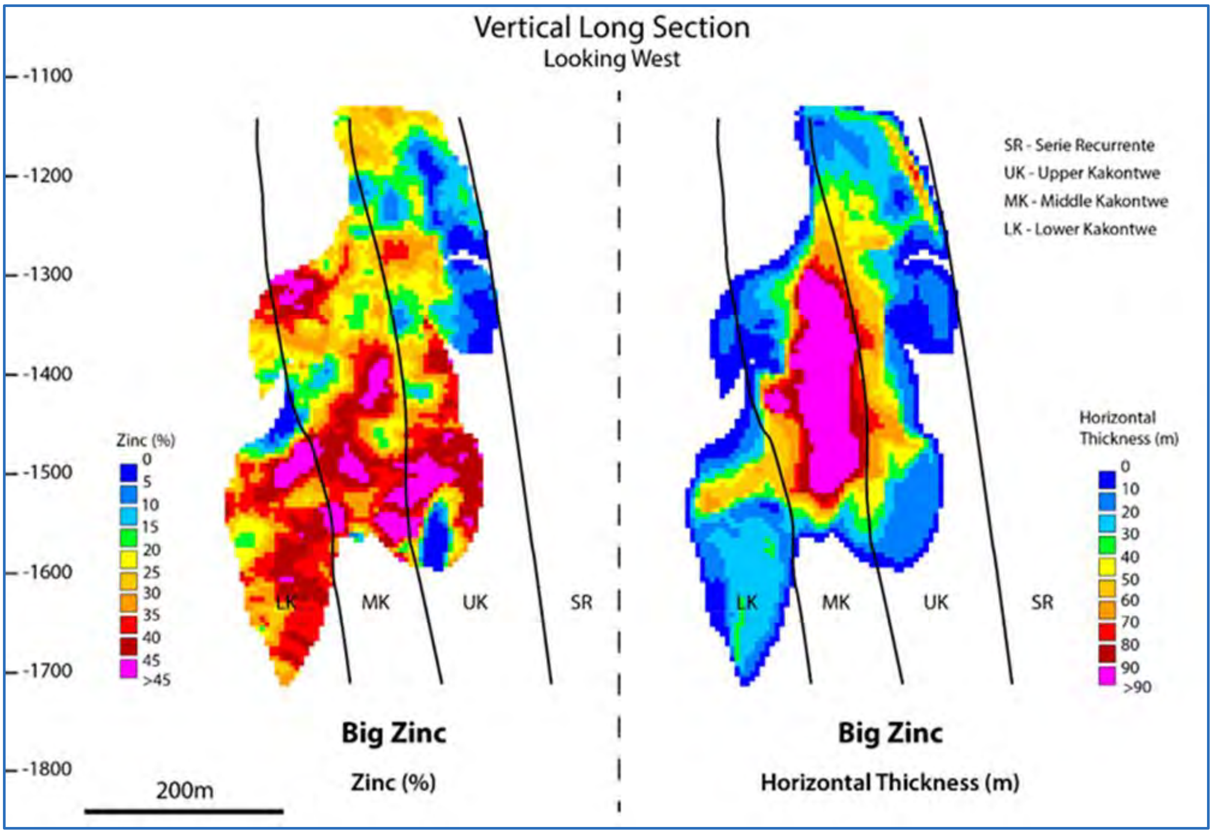
Localised internal barren to lower-grade 'stérile' zones occur and were defined by Gécamines on the visual basis of 7% Zn and/or 1% Cu cut-offs. Drill core from these zones was generally not preserved by Gécamines.

Figure 7.15 Mineralisation Intersected in Metallurgy Drillhole KPU104



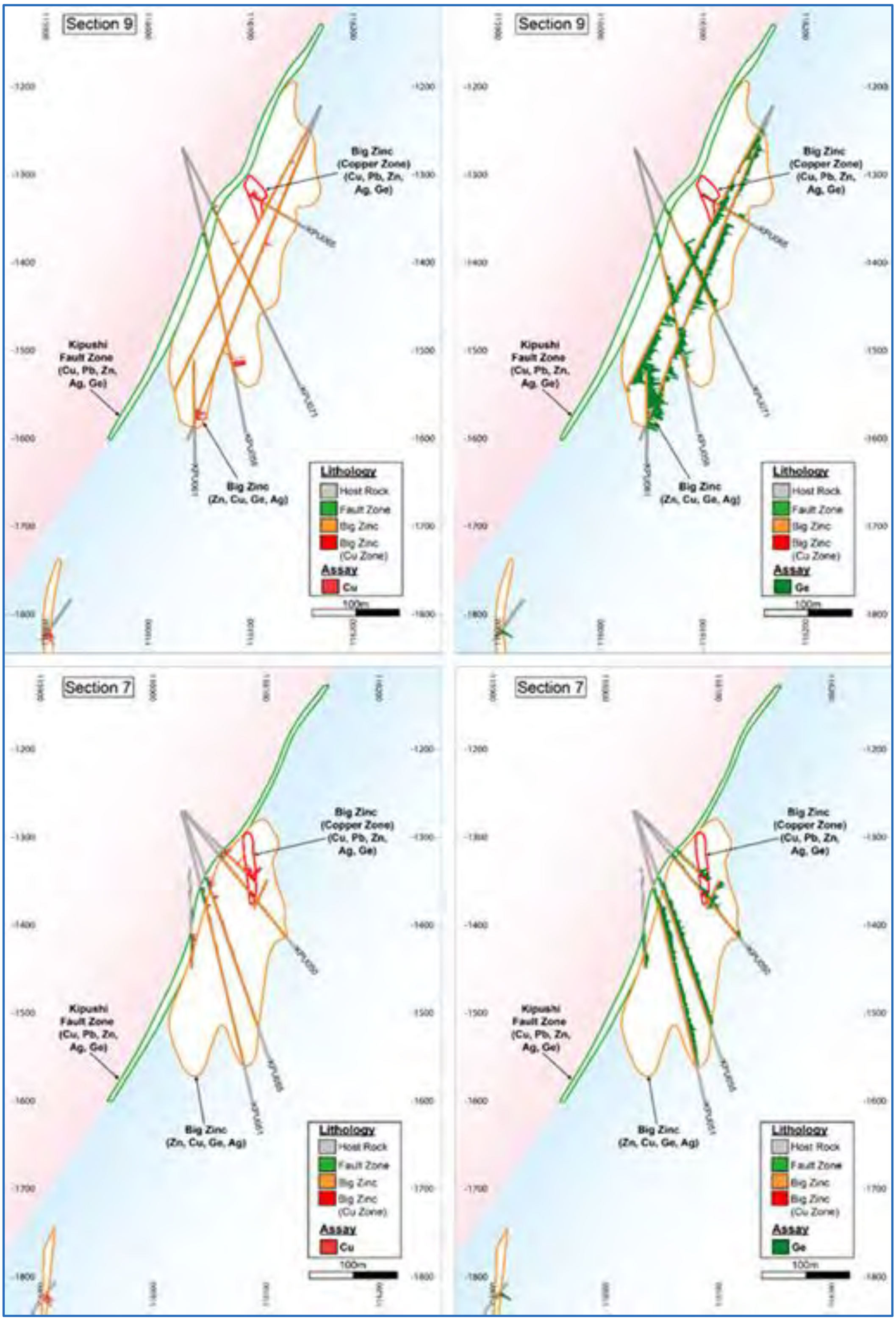
Ivanhoe Mines, 2018;
(A) Reddish iron-rich sphalerite and pyrite on the margins of the Big Zinc, (B) the dominant massive honey-brown sphalerite, (C) Copper rich mineralisation (chalcopyrite, tennantite, and silver sphalerite) of the Cu-Zn zone in the central part of the Big Zinc.

Figure 7.16 Vertical Long Section Looking West, Showing the Zinc Grade and the Thickness Plot, in Relation to the Stratigraphic Contacts



Ivanhoe Mines, 2018

Figure 7.17 Copper and Germanium Distribution Through the Big Zinc (Sections 7 and 9)



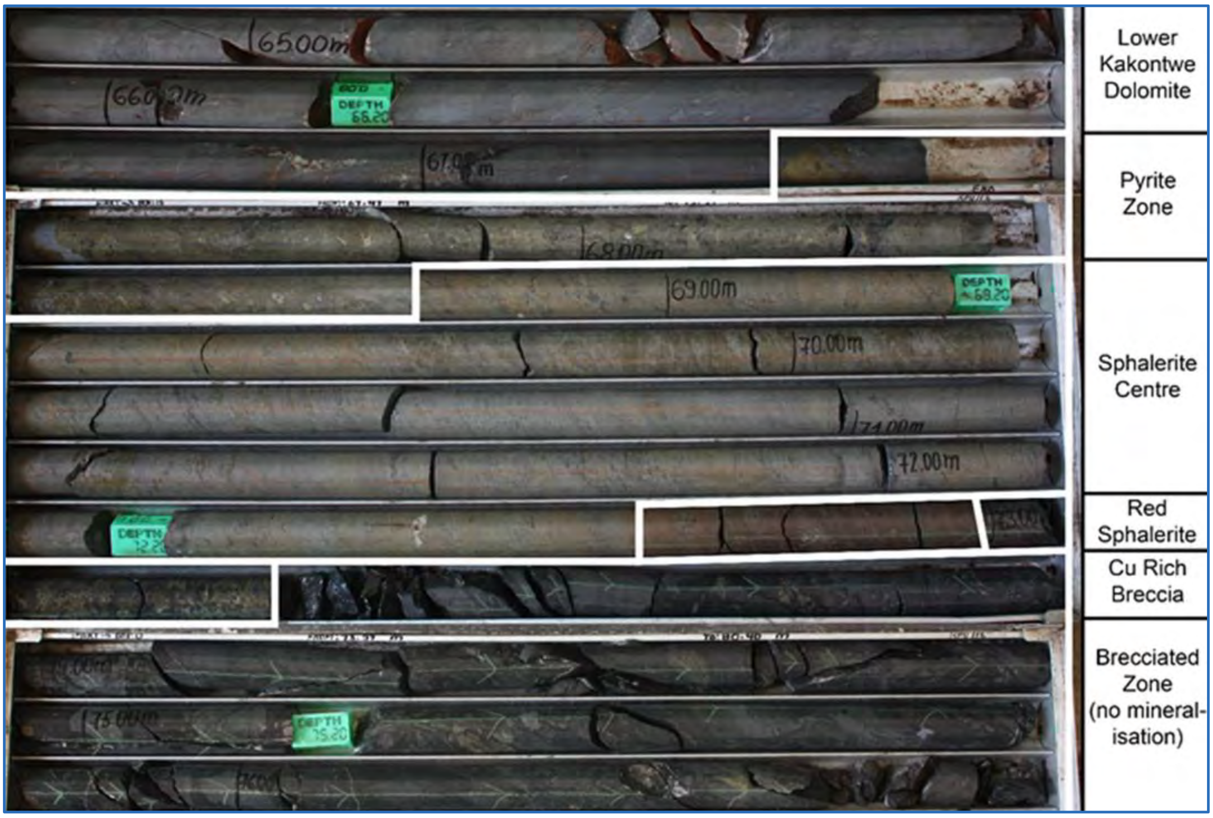
Ivanhoe Mines, 2018

7.4.3 Southern Zinc

The Southern Zinc mineralisation is a polymetallic mix of zinc and copper sulfides. This contrasts with the Big Zinc's massive honey-brown sphalerite zinc style mineralisation, with significantly more copper mineralisation contained in the Southern Zinc. It also has elevated silver, lead, arsenic and germanium values. The Southern Zinc displays two distinct mineralisation types: massive brown sphalerite, with minor chalcopyrite and massive silver sphalerite, with bornite and chalcopyrite. The mineralisation occurs along the contact of the Lower Kakontwe dolomites and the Fault Zone.

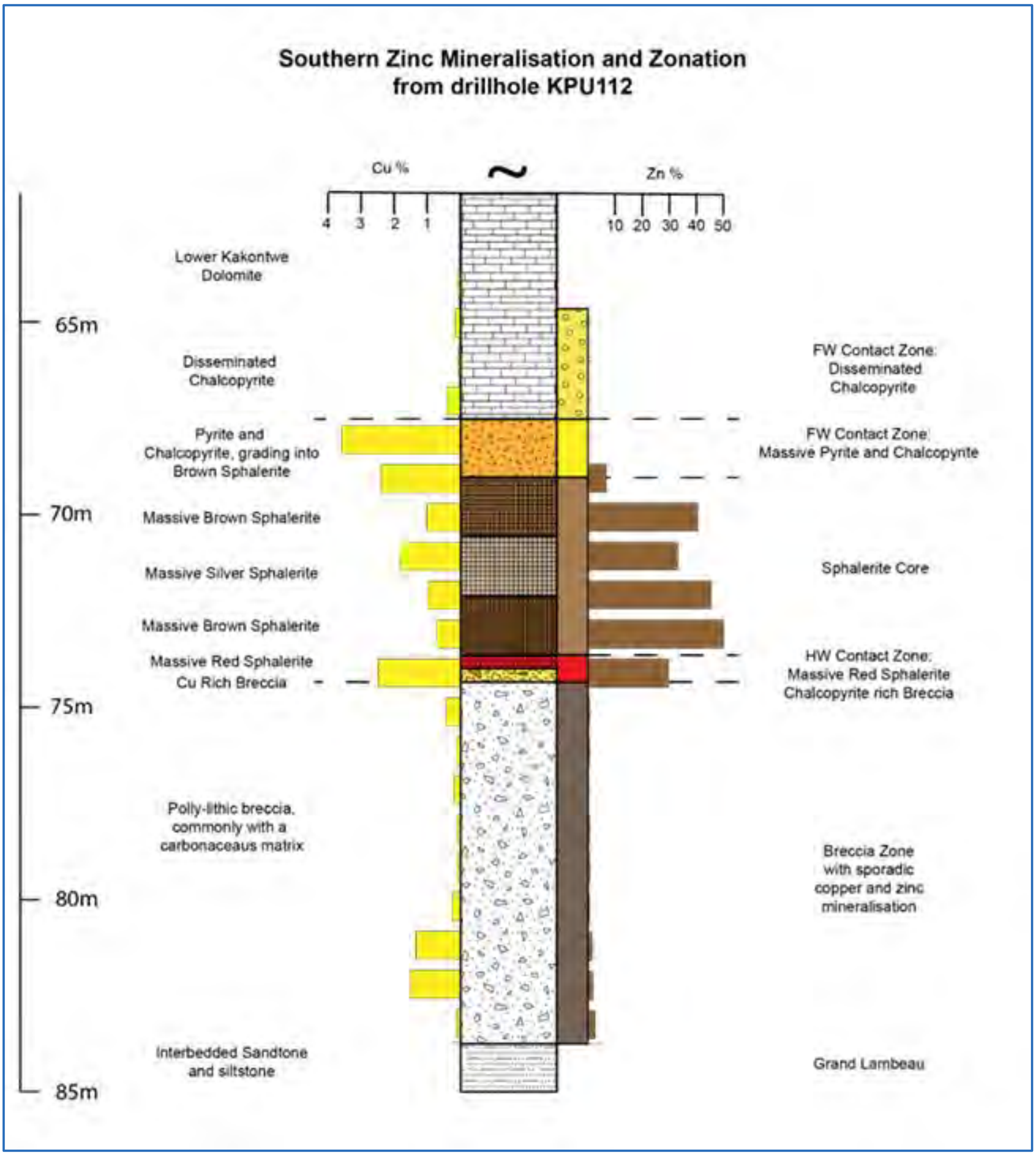
The Southern Zinc displays a definite mineralisation zonation. Disseminated chalcopyrite occurs within the footwall dolomites (not present in all the drillholes), followed by an abrupt change to massive sulfide mineralisation closer to the Fault Zone. The massive sulfides are zoned as follows; a pyrite contact zone, a massive sphalerite centre, followed by a red sphalerite zone immediately adjacent to the disseminated copper-rich Fault Zone (Figure 7.18 and Figure 7.19).

Figure 7.18 Southern Zinc Mineralisation Zonation from Drillhole KPU112



Ivanhoe Mines, 2018

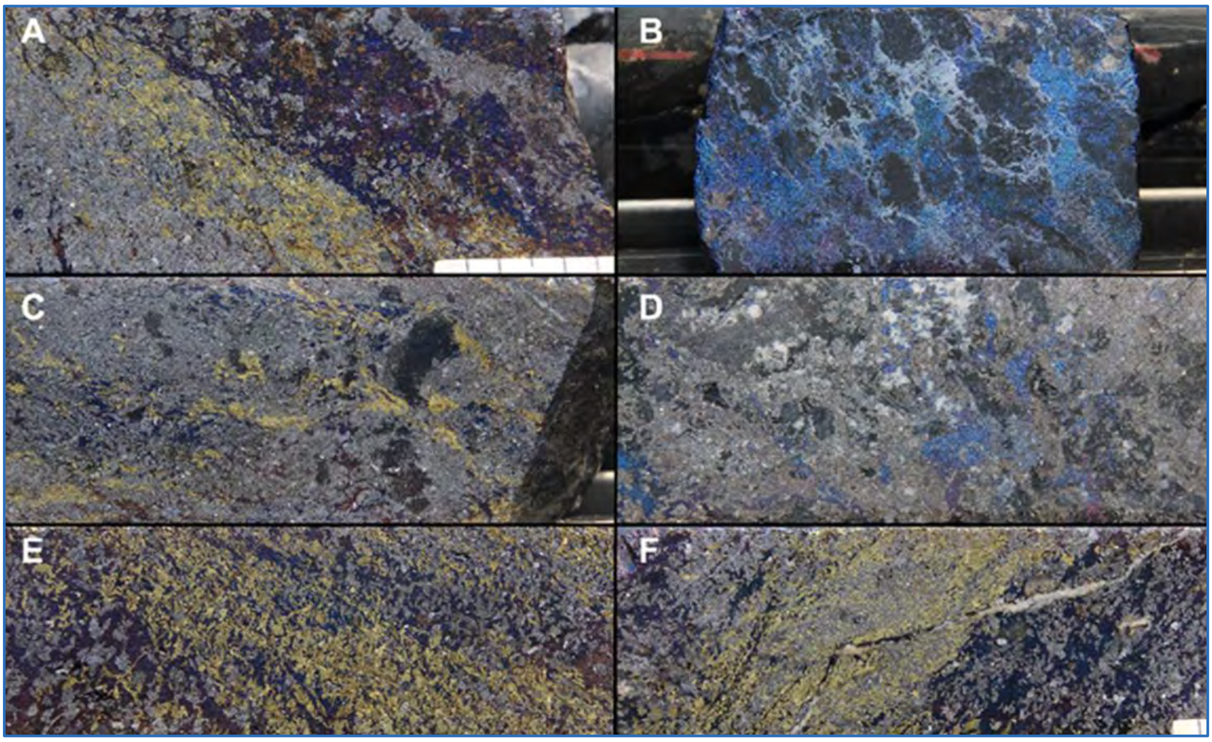
Figure 7.19 Idealised Southern Zinc Mineralisation Zonation



Ivanhoe Mines, 2018

There is a distinct zone in the Southern Zinc with silver sphalerite developed as opposed to the honey-brown sphalerite. This zone is also associated with elevated copper grades, and there appears to be an affinity for silver sphalerite and bornite to coexist. Bornite and chalcopyrite are both developed, but zones of massive bornite are common. Figure 7.20 shows some different examples of the silver sphalerite, bornite and chalcopyrite interaction.

Figure 7.20 Silver Sphalerite, Bornite and Chalcopyrite Mineral Assemblage from KPU125



Ivanhoe Mines, 2018;
Polymetallic mineral assemblage from KPU125; (A) Zonation from chalcopyrite to bornite (left to right) within silver sphalerite, (B) Bornite and silver sphalerite, surrounding dolomite clasts, (C) wispy chalcopyrite and bornite within silver sphalerite, (D) Bornite and silver sphalerite replacing white and black dolomite, (E and F) Bornite, chalcopyrite and silver sphalerite assemblages.

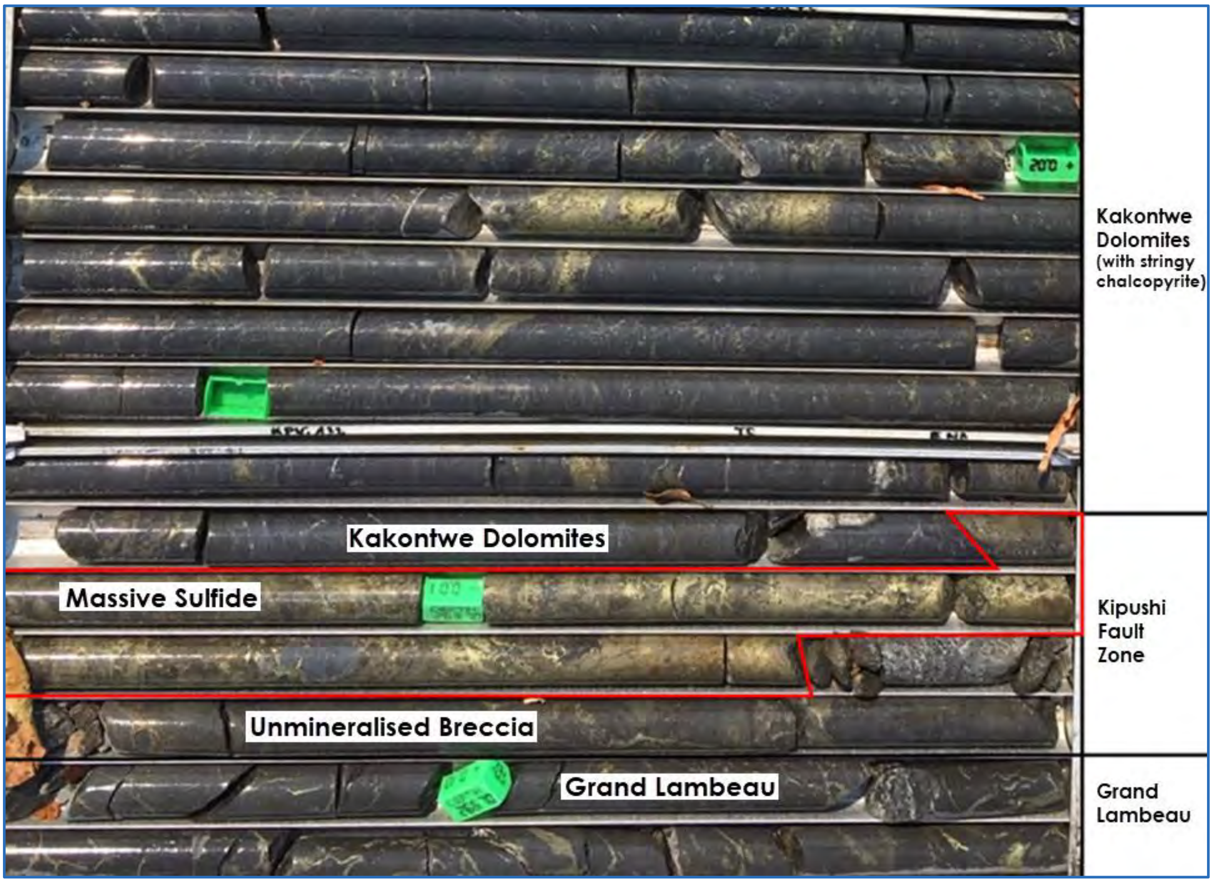
7.4.4 Kipushi Fault Zone

The Kipushi Fault Zone comprises Cu–Zn-Pb-Ag-Ge mineralisation developed along the steeply north-west dipping Kipushi Fault or reef edge (see Section 7.2.2), between the Grand Lambeau to the west and intact Nguba Group stratigraphy to the east. Mineralisation locally extends laterally as discordant offshoots into rocks of the Kakontwe and Série Récurrente Formations in the footwall to the Fault Zone and terminates to the south-west where it intersects the Grand Conglomérat (Mwale Formation).

The Fault Zone deposit forms an irregular tabular body over a strike length of approximately 600 m and variable thickness that narrows with depth (Figure 7.17). The thickness varies from approximately 1 m to more than 20 m, with typical thicknesses ranging from 5–10 m. Below 1,400 mL, the Big Zinc diverges from the zinc–copper-lead-rich Fault Zone deposit, as shown in Figure 7.12.

The Fault Zone features a diverse range of textures, lithologies, mineralisation styles and types. The grade is variable and shares the same southerly plunge as the Big Zinc and Southern Zinc, described in Section 7.4.1 and shown in Figure 10.3. Between approximately 1,200–1,350 mL, the Big Zinc mineralisation contacts the Fault Zone, where it is partially replaced with sphalerite and pyrite. The defined Southern Zinc, to a depth of 1,400 mL, is always seen to abut the Fault Zone (Figure 7.18 and Figure 7.19). High-grade portions of the Fault Zone include disseminated to semi-massive chalcopyrite mineralisation that extends into the Kakontwe dolomites immediately adjacent to the Fault Zone (Figure 7.21).

Figure 7.21 Fault Zone Copper Mineralisation Extending into the Middle Kakontwe Dolomites from Drillhole KPU132



Ivanhoe, 2018; The disseminated mineralisation in the Kakontwe dolomite, extends further than displayed in the figure, in a similar style of stringy disseminated chalcopyrite.

7.4.5 Copper Nord Riche

Discreet mineralised zones of patchy to massive chalcopyrite with minor sphalerite are focussed at the top of the Upper Kakontwe Formation (Figure 7.22), near its contact with the Série Récurrente Formation. This area is locally known as the Copper Nord Riche. Mineralisation in the Copper Nord Riche is significantly thicker than in the adjacent Série Récurrente. In the Copper Nord Riche, the mineralised zones are oblate and discordant, cutting down stratigraphy and thickening in closer proximity to the Kipushi Fault Zone, especially at the termination of the Upper Kakontwe against the Fault Zone (Figure 7.11 and Figure 7.23). Chalcopyrite intercepts frequently contain elevated silver (>100 ppm), arsenic (>5,000 ppm), and molybdenum (>100 ppm) associated with tennantite (Figure 7.23).

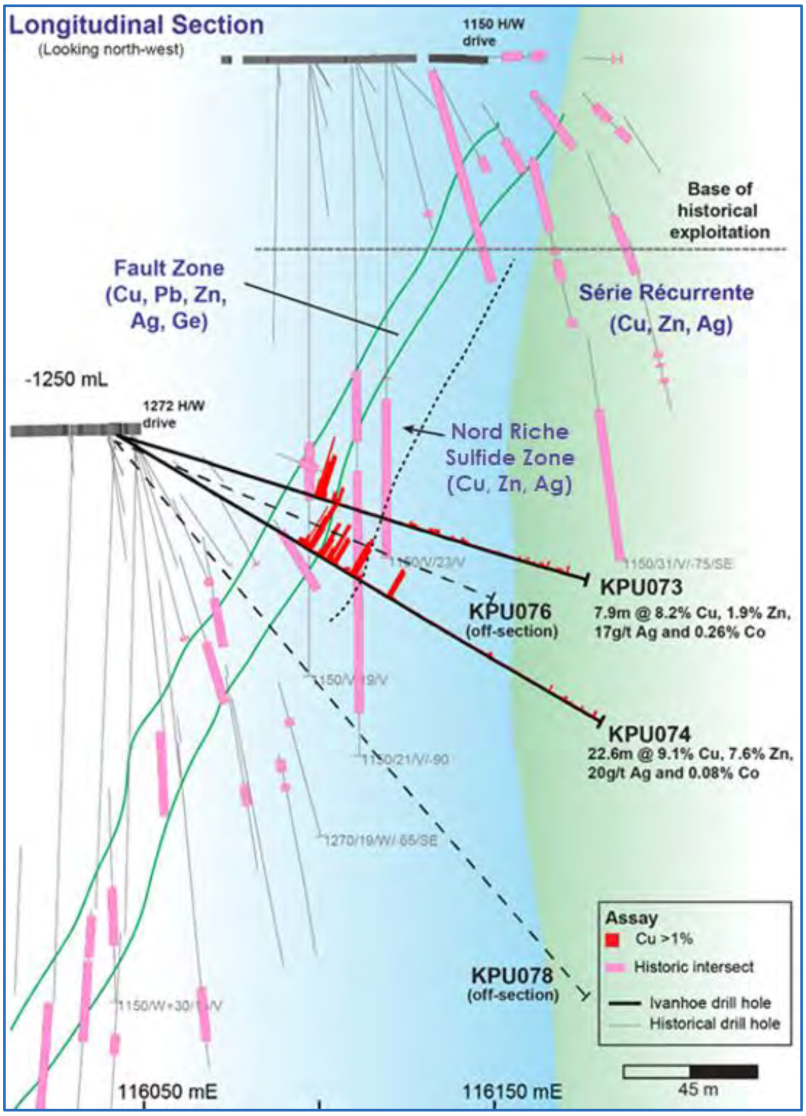
Replacement mineralisation in the Upper Kakontwe has an association with locally disrupted bedding. Parasitic folds in the plane of bedding plunging at steep angles, seem to localise mineralisation and replacement.

Due to a lack of suitable drill sites, the Copper Nord Riche has been incompletely explored below the previous workings. Planned deepening of the decline will provide opportunities to test this zone more systematically.

Figure 7.22 Drillhole KPU105 Showing Massive Chalcopyrite Mineralisation in the Upper Kakontwe near the Northern Limit of the Fault Zone

Ivanhoe Mines, 2018

Figure 7.23 Longitudinal Section at the Northern End of the Fault Zone Looking North-west and Showing the Fault Zone, Nord Riche and Série Récurrente Mineralisation, Together with Historical and Some Recent Drilling



Ivanhoe Mines, 2018

7.4.6 Série Récurrente

There are two types of mineralisation styles that occur within the Série Récurrente (Figure 7.11):

- The first and most common is the disseminated chalcopyrite–bornite mineralisation within alternating siltstones and dolomite beds of the Série Récurrente Formation.
- The second is a massive sulfide pod, comprised predominantly of chalcopyrite, but with minor tennantite and sphalerite, that occurs within the Upper Kakontwe dolomites, directly adjacent to the Série Récurrente Formation.

7.4.6.1 Disseminated Série Récurrente Mineralisation

The disseminated mineralised zone (Figure 7.24) extends at least 150 m eastward along strike, from the Fault Zone. Copper grades are generally around 1–2%, and this mineralisation extends from the Upper Kakontwe Formation contact, approximately 20 m into the Série Récurrente Formation, gradually diminishing with increasing distance from the contact. Bornite tends to become more abundant than chalcopyrite northwards from the contact. The bornite mineralisation tends to be localised in dolomite beds whereas chalcopyrite dominates in the siltstone beds, where it occurs with trace Mo and Re.

Mineralisation is best developed in the siltstone beds, where it occurs as discrete 2–5 mm thick discontinuous veinlets or lenticles parallel or subparallel to foliation/bedding (Figure 7.25). These veinlets or lenticles are always associated with quartz/carbonate of a coarser grain size than the siltstone host, and commonly exhibit a strong structural control. Strain accommodated along bedding planes in the siltstone appears to have deformed earlier veinlets. Mineralisation in dolomite is also vein-hosted, but without the strong structural control seen in the deformed siltstone.

Figure 7.24 Typical Colour Variation in the Série Récurrente Between Dolomite (Purple) and Siltstone (Green)

Ivanhoe Mines, 2015

Figure 7.25 Blebby and Disseminated Chalcopyrite in Série Récurrente Siltstone at 148 m in Drillhole KPU074

Ivanhoe Mines, 2015; Both Mineralisation and Bedding are Deformed by Parasitic Folds

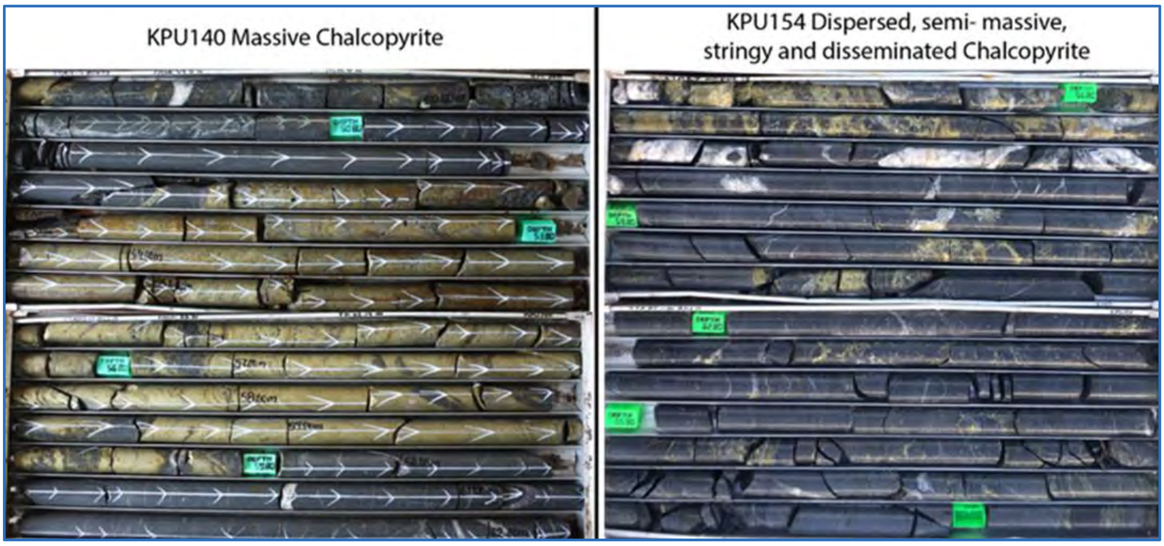
7.4.6.2 Massive Sulfide Pod

The massive sulfide pod is located within the Upper Kakontwe dolomites, approximately along the Upper Kakontwe – Série Récurrente contact (Figure 7.11), although the strike and dip of the body is discordant to stratigraphy. Along strike to the east the zone transgresses into the Upper Kakontwe; similarly down plunge the body moves progressively away from this contact and entirely into the Upper Kakontwe (Figure 10.6).

The body is generally thickest in the centre and thins towards the extremities. Concordantly the mineralisation is developed as massive sulfides in the centre and transforms toward the edge and western extremities into disseminated and stringy sulfides, with only a small, massive sulfide portion. The different mineralisation styles are shown in Figure 7.26.

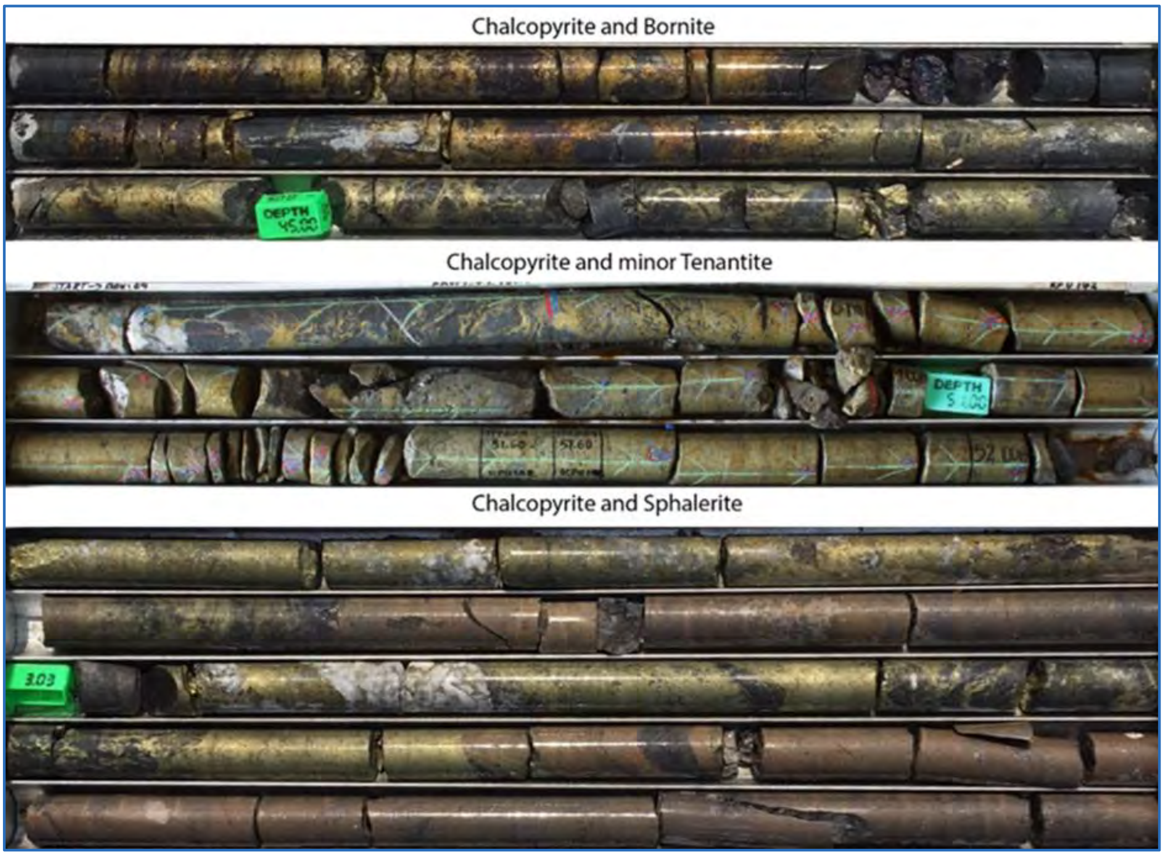
There is also a distinct mineral zonation evident within the sulfide body, chalcopyrite dominates, but there is a clear vertical zonation; a mixture of chalcopyrite and bornite up-plunge; a central portion of mostly chalcopyrite with some minor tennantite; and down-plunge, a mixture of chalcopyrite and mostly red-brown sphalerite (Figure 7.27).

Figure 7.26 Mineralisation Style within the High-Grade Pod



Ivanhoe Mines, 2018

Figure 7.27 Série Récurrente Mineralisation Zonation from Up-plunge, Central Portions, and Down-plunge



Ivanhoe Mines, 2018

8 DEPOSIT TYPES

This section has not been changed from the Kipushi 2019 Resource Update and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Kipushi 2019 Resource Update.

The mineral deposits at Kipushi are an example of carbonate-hosted copper–zinc-lead mineralisation hosted in pipe-like replacement and tabular zones. This deposit type tends to form irregular, discordant mineralised bodies within carbonate or calcareous sediments, forming massive pods, breccia/fault-like fillings, and stockworks (Trueman, 1998). They often form pipe-like to tabular deposits strongly elongate in one direction. Zinc-lead rich zones can project from the main zone of mineralisation as replacement bodies parallel to bedding, as is the case at Kipushi.

This deposit type is associated with intracratonic platform and rifted continental margin sedimentary sequences which are typically folded and locally faulted (Cox and Bernstein, 1986). The host carbonate sediments were deposited in shallow marine, inter-tidal, salt flat, lagoonal, or lacustrine environments and are often overlain unconformably by oxidised sandstone-siltstone-shale units. The largest deposits are Neoproterozoic in age and occur within thick sedimentary sequences.

No association with igneous rocks is observed. Mineralisation forms as fault or breccia filling, and massive replacement mineralisation with either abundant diagenetic pyrite or other source of sulfur (e.g. evaporates) acting as a precipitant of base metals in zones of high porosity and fluid flow. The presence of bitumen or other organic material is indicative of a reducing environment at the site of metal sulfide deposition. Deposits are usually coincident with a zone of 106rebody106zation. Pre-mineralisation plumbing systems were typically created by karsting, faulting, collapse zones as a result of evaporate removal, and/or bedding plane aquifers and were enhanced by volume reduction during 106rebody106zation, ongoing carbonate dissolution and hydrothermal alteration (Trueman, 1998). It is considered that oxidised diagenetic fluids scavenged metals from clastic sediments from a source area with deposition in open spaces in reduced carbonates, often immediately below an unconformity.

A number of epigenetic copper–zinc-lead massive sulfide deposits are hosted in deformed platform carbonates of the Lufilian Arc. In the Democratic Republic of Congo (DRC), these are mostly hosted in carbonate units of the Kaponda, Kakontwe, Kipushi, and Katete (Série Récurrente) Formations of the Nguba Group. These units are characterised by shallow water marine carbonates, predominantly dolomitic, associated with organic-rich facies (Kampunzu, et el., 2009). Although most of these are relatively small, they include the major deposits of Kipushi and Kabwe which occur as irregular pipe-like bodies, as well as lenticular bodies subparallel to stratigraphy. They are thought to be associated with collapse breccias and faults (Kampunzu, et al., 2009) and tend to be surrounded by silicified dolomite. These carbonate-hosted copper–zinc-lead deposits tend to contain important by-products of silver, cadmium, vanadium, germanium, and gallium.

Fluid inclusion and stable isotope data from Kipushi indicate that hydrothermal metal-bearing fluids evolved from formation brines during basin evolution and later tectonogenesis (Kampunzu, et el., 2009). Mineralised fluid migration occurred mainly along major thrust zones and other structural discontinuities such as breccias, faults, and karsts within the Katangan Supergroup resulting in metal sulfide deposition within favourable structures and reactive carbonate sequences. In the case of the Big Zinc and Southern Zinc, massive sphalerite mineralisation is a result of extensive replacement of the host carbonates.

Other examples of this model include Tsumeb and Kombat in Namibia, Ruby Creek, and Omar in Alaska, Apex in Utah, and M'Passa in the Republic of Congo.

9 EXPLORATION

No other relevant exploration work, other than drilling, has been carried out by Kipushi Corporation SA (KICO) on the Kipushi Project.

10 DRILLING

This section has not been changed from the Kipushi 2019 Resource Update and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Kipushi 2019 Resource Update.

10.1 Historical Drilling

10.1.1 Drilling Methodology

La Générale des Carrières et des Mines (Gécamines') drilling department (Mission de Sondages) historically carried out all drilling. Underground diamond drilling involved drilling sections spaced 15 m apart along the Kipushi Fault Zone and Big Zinc and 12.5 m apart along the Série Récurrente, with each section consisting of a fan of between four and seven holes (Figure 10.1), the angle between the holes being approximately 15° (Kelly et al., 2012). Sections are even-numbered south of Section 0 and odd-numbered to the north. Drilling was completed along the Kipushi Fault Zone from Section0–19 along a 285 m strike length including a 100–130 m strike length which also tested the Big Zinc and a 50–200 m interval that tested the Southern Zinc.

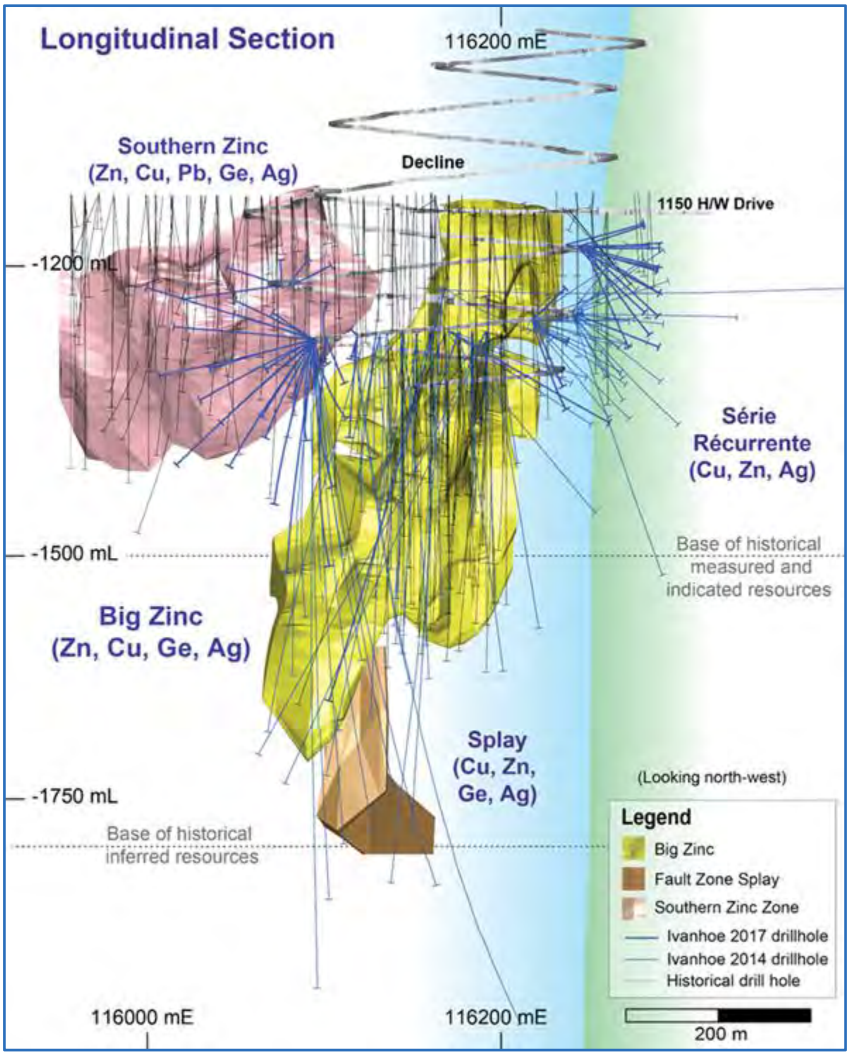
Drill core from 49 of the 60 holes drilled from 1,272 mL that intersected the Big Zinc are stored under cover at the Kipushi Mine. The retained half core is in a generally good condition and is mostly BQ in size with subordinate NQ core. In general, only mineralised intersections were retained, with only minor barren or 'stérile' zones preserved in the core trays. The 'stérile' zones were based on a visual cut-off of 1% Cu and 7% Zn, and where preserved are observed to contain variable disseminated and vein hosted sphalerite mineralisation.

The drilling methodology is described in Kelly et al., (2012) where they noted that some of the drill log sheets contained missing information. On completion of each drillhole, collar and downhole surveys were conducted, and the following information was recorded on drill log sheets:

- Hole number, with collar location, length, inclination, and direction.
- Start and completion dates of drilling.
- Collar location (X, Y, Z coordinates), azimuth, and inclination.
- Hole length and deviation.
- Core lengths and recoveries.
- Geological and mineralogical descriptions (often simplified).
- Assay results.
- Hydrology and temperature.

The Historic Gécamines drillholes have been used in conjunction with the Kipushi Corporation SA (KICO) drillholes in the resource estimation, described in Section 14. Gécamines sampling tended to be based on lengths representing mineable zones, with little attention paid to geology and mineralisation (Kelly et al., 2012).

Figure 10.1 Long Section of the Big Zinc Showing the Projection of Drillhole Traces for Gécamines and KICO Drillholes



Ivanhoe Mines, 2018

10.1.2 Drillhole Database

Hardcopy information from the log sheets were transferred into a digital database, with the data being encoded by a local team. The following data were captured:

- Drillhole ID, collar coordinates, azimuth, inclination, length, core recovery, date of completion, and remarks.
- Assay results for Zn, Cu, Pb, S, Fe, and As.
- Geological and mineralisation log, as standardised simple codes.
- Downhole survey data.
- Hydrology data.

Validation of the captured data was undertaken. A total of 762 holes for a total of 93,000 m and 7,500 samples for a total of 51,500 assays were captured.

In addition, The MSA Group (Pty) Ltd (MSA) undertook a data capturing exercise of drillholes from digital scans of hard copy geological logs which is described further in Section 14.

10.2 KICO Drilling

All work carried out during the KICO underground drilling project was performed according to documented standard operating procedures for the Kipushi Project. These procedures covered all aspects of the programme including drilling methodology, collar and downhole surveying, metre marking, oriented drill core mark-ups, core photography, geological and geotechnical logging, and sampling.

10.2.1 Drilling Methodology

The Kipushi Mine was placed on care and maintenance in 1993 and flooded in early 2011 due to a lack of pumping maintenance over an extended period. Following dewatering and access to the main working level in December 2013, an original 25,400 m underground drilling programme was carried out by KICO between March 2014 and October 2015. A subsequent 9,700 m drilling campaign was carried out from May–October 2017.

24 April 2018 is the cut-off date for data included in the Mineral Resource.

The original drilling was designed to confirm and update Kipushi's Historical Estimate and to further expand the drilled extents of mineralisation along strike and at depth. Specifically, the objectives of the first drilling programme were to:

- Conduct confirmatory drilling to validate the Historical Estimate within Kipushi's Big Zinc and Fault Zone and qualify them as current Mineral Resources prepared in conformance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) standards as required by National Instrument 43-101.
- Conduct extension drilling to test the deeper portions of the Big Zinc and Fault Zone below 1,500 mL.
- Test for deeper extensions to the Big Zinc by drilling from the 1,272 mL hanging wall drive and from various locations on the footwall decline.
- Conduct exploration drilling to test areas that have not been previously evaluated, such as the deeper portions of the Fault Zone and extensions to the high-grade copper mineralisation of the mine's Copper Nord Riche.
- Gain an improved understanding of geology and controls on mineralisation.

The 2017 drilling campaign focussed on the following:

- Metallurgical and geotechnical drilling, to be used in the testwork for Section 13 and Section 16 respectively (Table 10.2).
- Confirm and expand the Southern Zinc, with the aim to upgrade the Inferred Mineral Resource into an Indicated Mineral Resource.

- To better understand the link between the Big Zinc and Southern Zinc mineralisation.
- Expansion drilling in the Série Récurrente, further delineating the high-grade pod.
- Target the Nord Riche and investigate the potential mineralisation close to the 1,272 mL hanging wall drive.

Underground drilling of the various mineralised zones were carried out from the footwall ramp and the hanging wall drive on 1,272 mL. Drilling at the project was undertaken by Major drilling SPRL from 1 March 2014 until the end of September 2014 when Titan Drilling Congo SARL took over diamond drilling operations. Titan Drilling were used again in the 2017 drilling campaign operating two Boart Longyear LM90 electro-hydraulic underground drill rigs.

Drilling was carried out on the same 15 m spaced sections used by Gécamines and comprised twin holes, infill holes and step-out exploration holes. Drilling on each section comprised a fan of between four and seven holes. The angle between the holes was ± 15º. KICO drilled the different targets from different locations. The northern part of the Big Zinc was drilled from the 1,272 mL hanging wall drill drive along the Fault Zone. The southern portion of the Big Zinc and the Southern Zinc was drilled from the decline below 1,272 mL, while the Série Récurrente drilling was performed from two levels in the decline, ~1,180 mL and ~1,250 mL.

Drilling was mostly NQ-TW (51 mm diameter) size with holes largely inclined downwards at various orientations to intersect specific targets within the Big Zinc, Fault Zone, Copper Nord Riche and Série Récurrente (Figure 10.1). Aside from the deeper parts of the Fault Zone where intersections are up to 100 m apart, the remainder of the mineralised intersections across all the different zones are between 10 m and 50 m apart.

As at the effective date of this report, a total of 157 holes had been drilled for 34,843 m including 57 holes that intersected the Big Zinc. The drilling breakdown of the two KICO campaigns is shown in Table 10.1, where the different targets are highlighted. The metallurgy and geotechnical drilling are included in Table 10.1, but also tabulated separately in Table 10.2.

Table 10.1 Underground Drilling Summary

Drilling Target	Year 2014/15				Total 2017	
	Drillholes	Metres	Drillholes	Metres	Drillholes	Metres
Big Zinc	56	18,864	3	980	59	19,844
Southern Zinc	1	258	30	6,010	31	6,268
Série Récurrente	37	4,751	16	1,398	53	6,149
Kipushi Fault Zone	2	452	0	–	2	452
Nord Riche	1	361	6	1,058	7	1,419
Stratigraphy	1	453	0	–	1	453
Footwall Exploration	0	–	4	258	4	258
Total	98	25,140	59	9,704	157	34,843

Table 10.2 2017 Metallurgy and Geotechnical Drilling Programme Breakdown

Drilling Target	Drillholes	Metres
Big Zinc	2	801
Southern Zinc	2	290
Série Récurrente	2	109
Nord Riche	1	201
Total	7	1,200

10.2.2 Core Handling

Drilling was undertaken and core recovered using standard wireline drilling. Core was carefully placed in aluminium core trays in the same orientation as it came out of the core barrel. Core trays were marked with the drillhole number, the start and end depths, a sequential tray number, and an arrow indicating the downhole orientation.

Core trays were delivered from underground to the core storage facility at the mine site.

10.2.3 Core Recovery

Core recovery was determined prior to geological logging and sampling. Standard core recovery forms were usually completed for each hole by the technician or geologist. Core recovery was also measured by the driller and included in drilling records.

Core recovery averaged 99.14% and visual inspection by the MSA's Qualified Person (QP) confirmed the core recovery to be excellent.

The Gécamines drillhole cores are in variable condition having been stored for long periods of time and moved around on occasions. No core recovery data are available from the original Gécamines records.

10.2.4 Collar and Downhole Surveys

All the KICO drillhole collars have been surveyed by a qualified surveyor. The surveyor was notified of the anticipated time of the rig move to ensure proper mark-up of the hole, and to be on site to monitor the positioning of the rig.

Gécamines collars were located in a local mine grid coordinate system. The mine grid coordinates were converted to Gaussian coordinates and validated against the surveys of the underground workings.

Downhole surveys were completed for all the KICO holes, with the majority surveyed at either 3 m or 5 m intervals. A few holes were surveyed at 30 m intervals. The KICO holes were surveyed using a Reflex EZ-SHOT downhole survey tool. As a check on accuracy and precision on this method, 13 holes were also surveyed using a Gyro Sealed Probe downhole survey instrument. No significant discrepancies were noted between the EMS and Gyro tools.

Downhole surveys are available for many of the Gécamines drillholes and were generally surveyed at 50 m downhole intervals. No details are available regarding the survey instruments used. Where no downhole survey data are available for a drillhole, the collar survey inclination and azimuth were used as the downhole survey.

10.2.5 Geological Logging

Standard logging methods, geological codes, and sampling conventions were established prior to and implemented throughout the project. All of the drillholes were geologically logged by qualified geologists employed by KICO. For the first 14 holes (KPU001 to KPU014) logging of lithology, alteration, mineralisation, and structure were recorded on standardised paper templates and then captured and validated on import into the MS Access database. From hole KPU015 onwards, all logging was done directly into MS Access. All geotechnical logging was done directly into MS Access.

Prior to sampling, drill cores were photographed both wet and dry up until KPU119, thereafter only wet photographs were taken.

A portable Niton XRF analyser was used to provide an initial estimate, on a metre by metre basis, of the concentrations of the more important elements present in the drill core.

10.2.6 Results

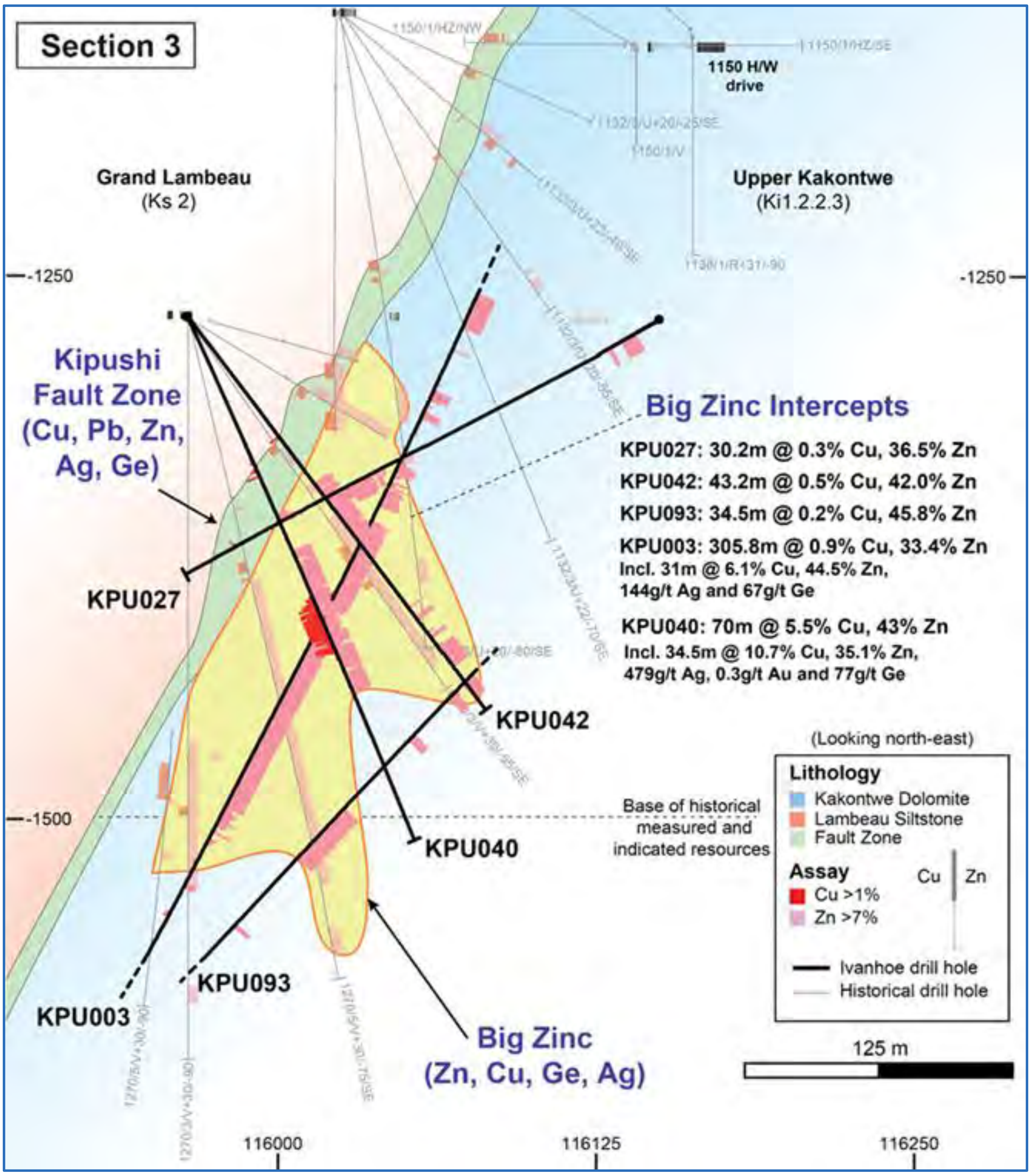
The KICO drilling focussed on the zinc dominant targets of the Big Zinc and Southern Zinc, as well as the copper dominant zones of the Fault Zone, the Nord Riche and the Série Récurrente. The deep extension of the Kipushi Fault Zone and the associated Splay have also intersected copper and zinc mineralisation.

10.2.6.1 Big Zinc

Drilling confirmed substantial widths and zinc grades within the Big Zinc and identified a high-grade copper and precious metals zone within the Big Zinc.

Figure 10.2 shows drill section 3, illustrating the KICO drilling results within the Big Zinc and Fault Zone.

Figure 10.2 Drill Section 3 Showing Drillholes through the Big Zinc

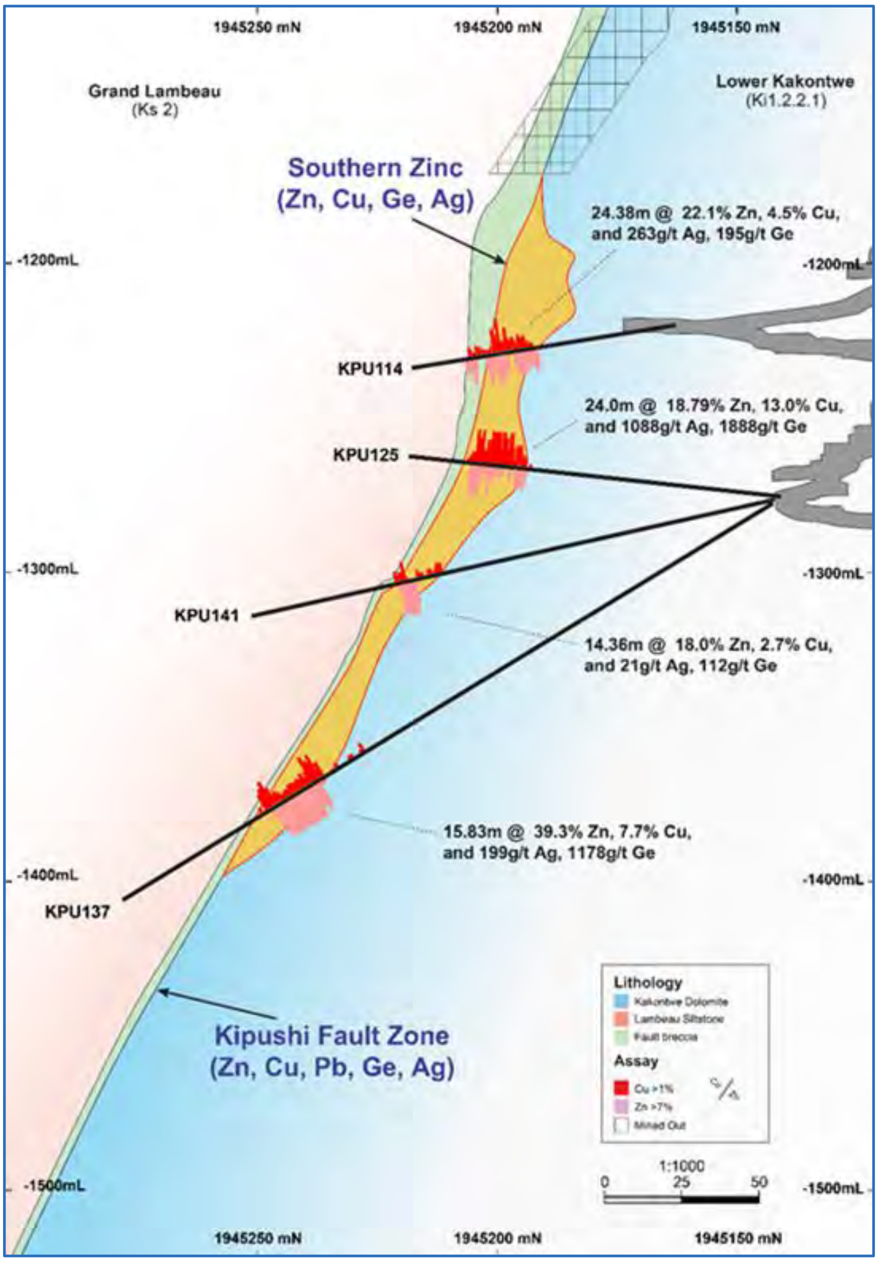


Ivanhoe Mines, 2018

10.2.7 Southern Zinc

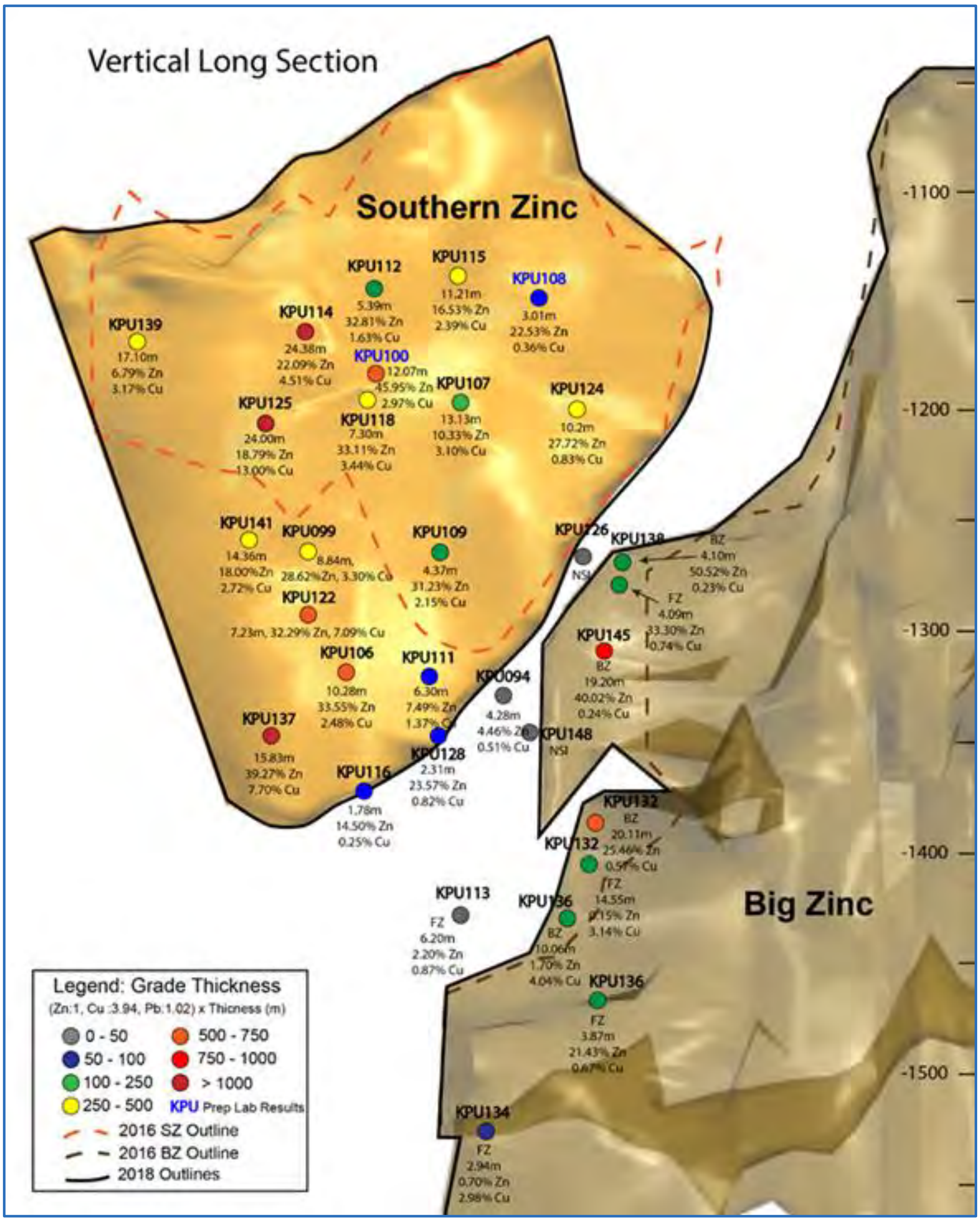
The polymetallic Southern Zinc is located south of the Big Zinc, abutting the Fault Zone (Figure 7.14). The zinc grade is lower when compared to the Big Zinc; while comparably, most of the other elements are elevated, including copper, silver, lead, arsenic, cadmium, and germanium. This increase appears to correlate to the occurrence of silver sphalerite within this zone. A cross-section and long section in Figure 10.3 and Figure 10.4 show the results.

Figure 10.3 Southern Zinc Cross-Section



Ivanhoe, 2018; The thickness is shown as true thickness.

Figure 10.4 Expanded Southern Zinc and Big Zinc Vertical Long Section



Ivanhoe Mines, 2018

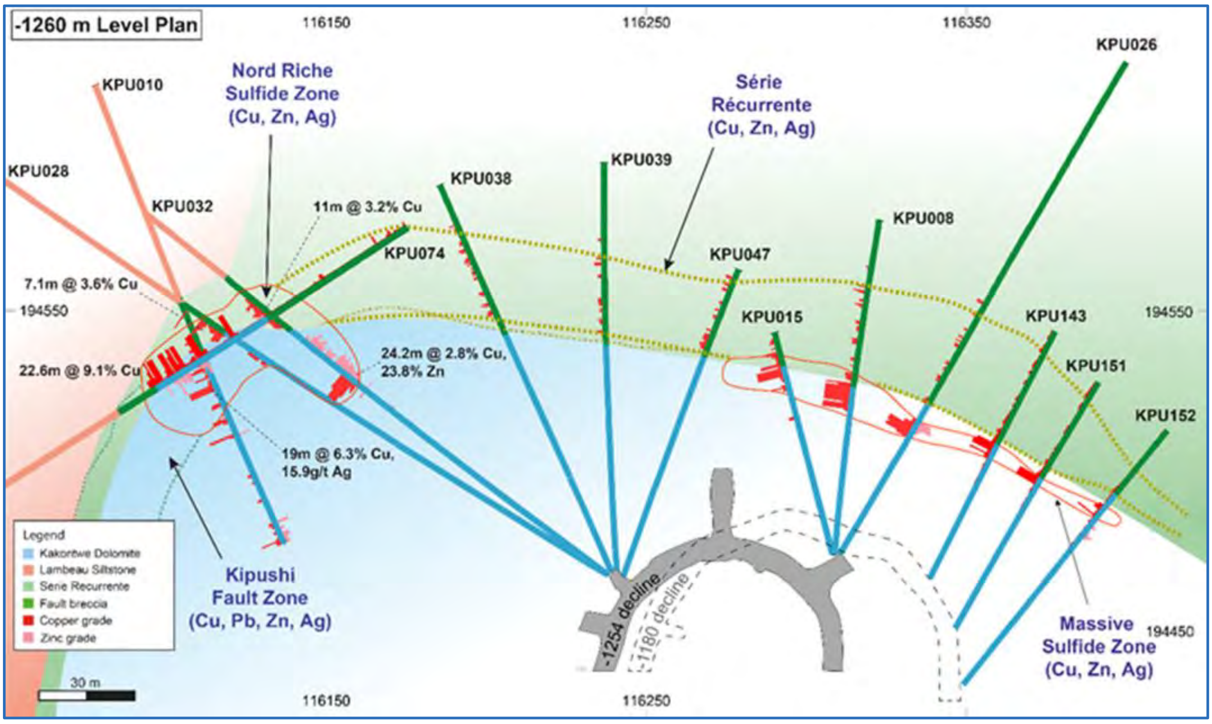
10.2.8 Copper Rich Zones of the Nord Riche, Série Récurrente, Fault Zone and the Splay

A plan projection of KICO drilling in the Copper Nord Riche and Série Récurrente is shown in Figure 10.5. Holes were drilled to test interpreted down-plunge extensions below the level of historical mining in the Copper Nord Riche area. These holes intersected zones of disseminated and massive sulfides (chalcopyrite and sphalerite).

The Série Récurrente contains a westerly-plunging lense of high-grade copper-rich massive sulfide, as described in Section 7.4.6.2, that extends from the Série Récurrente into the Upper Kakontwe. Drilling by Gécamines intersected this zone up-plunge but it was not mined. KICO drilling in both the 2014 and 2017 campaigns intersected this massive mineralisation, with the extent shown in Figure 10.6. The mineralisation appears to pinch and swell, therefore, potential extension to the east is still possible.

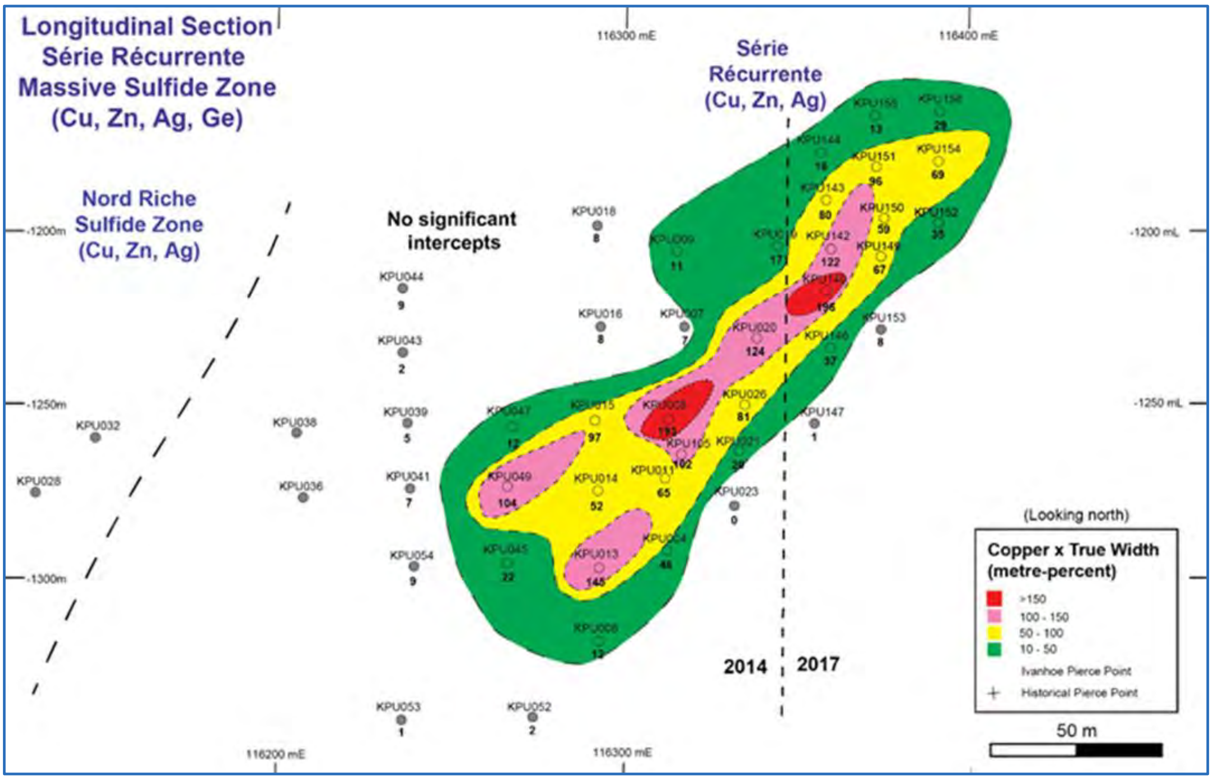
The disseminated style of 1–2% Cu mineralisation within the Série Récurrente can also be seen in Figure 10.5.

Figure 10.5 Drill Plan of 1,260 mL Showing KICO Drilling in the Copper Nord Riche and Série Récurrente



Ivanhoe Mines, 2018

Figure 10.6 Long Section of the High-Grade Série Récurrente Pod

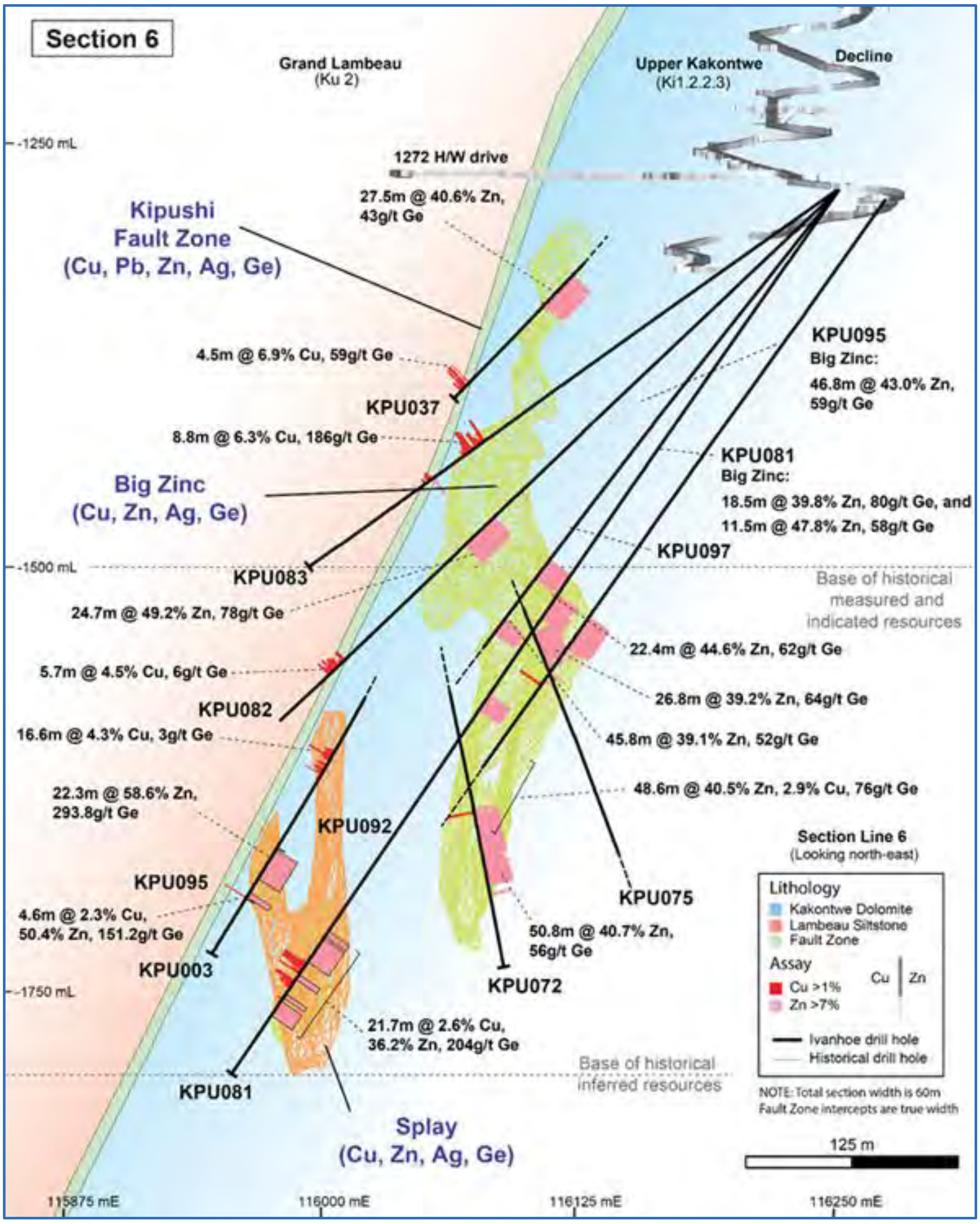


Ivanhoe Mines, 2018

A polymetallic zone has also been defined at depth below the Big Zinc, termed the Splay. It appears to have a structurally controlled plunge diverging from the Fault Zone into the Kakontwe Dolomites (Figure 10.7). The mineralisation includes significant copper and zinc grades, as well as anomalously high germanium.

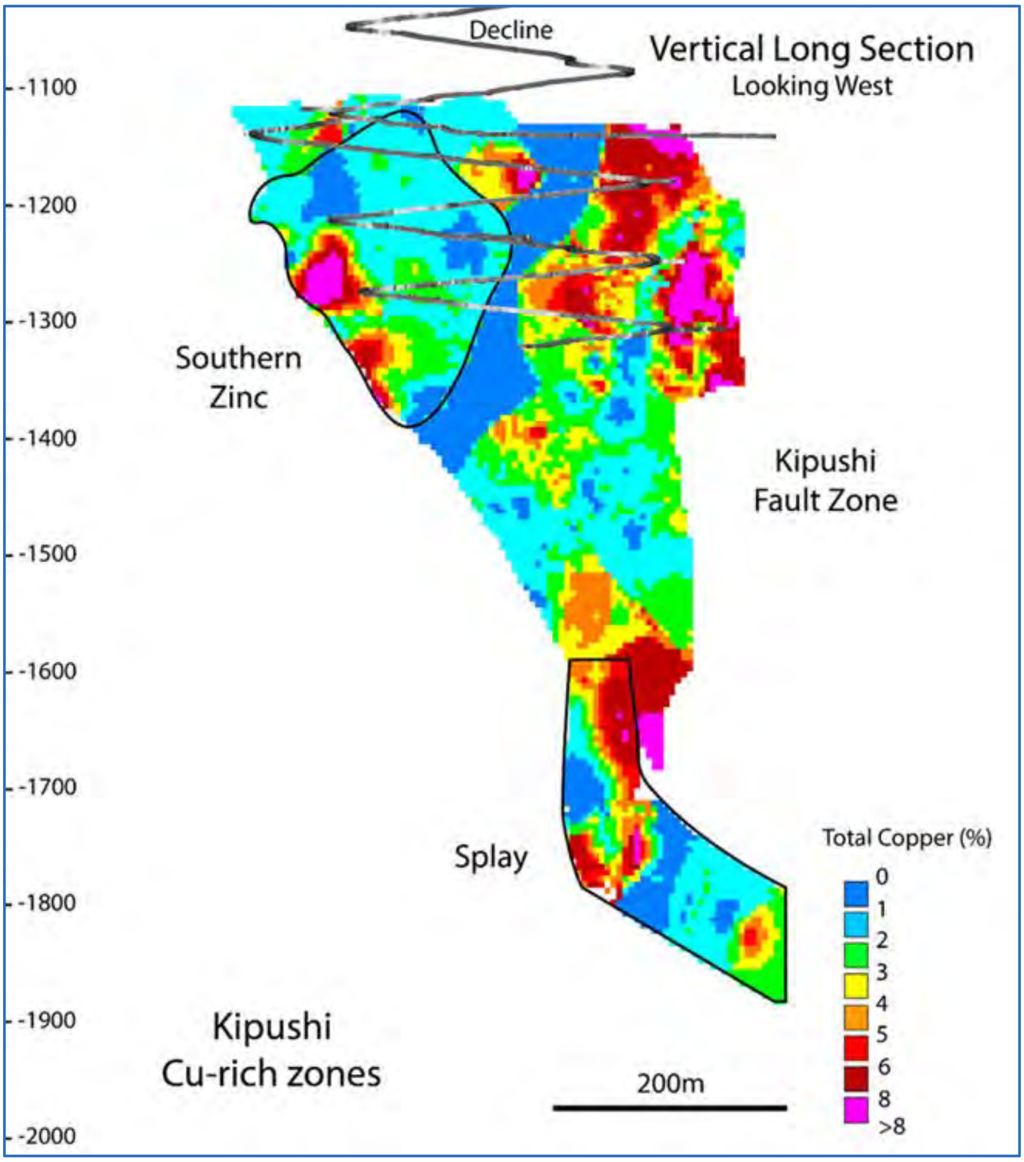
The copper dominant Fault Zone along with the associated copper mineralisation of the Southern Zinc and Splay zones are shown in the vertical long section in Figure 10.8. The southerly plunge to the mineralisation is clearly evident, with the two high-grade trends associated with the Big Zinc and Southern Zinc, separated by a low-grade zone.

Figure 10.7 Schematic Drill Section 6 Showing Drillholes through the Big Zinc and the Fault Zone Splay



Ivanhoe Mines, 2018

Figure 10.8 Vertical Long Section of the Fault Zone, Showing Copper Mineralisation



Ivanhoe Mines, 2018

10.2.9 QP Comment

In the opinion of the MSA QPs, the quantity and quality of data collected in the KICO underground drilling programme, including lithology, mineralisation, collar, and downhole surveys, is sufficient to support Mineral Resource estimation. This is substantiated further as follows:

- Core recoveries are typically excellent.
- Drillhole orientations are mostly appropriate for the mineralisation styles at Kipushi and adequately cover the geometry of the various mineralised zones, although several deep holes intersect the Fault Zone and Fault Zone Splay at a narrow angle.
- Core logging meets industry standards and conforms to exploration best practice.
- Collar surveys were performed by qualified personnel and meet industry standards.
- Downhole surveys were carried out at appropriate intervals to provide confident 3D representation of the drillholes.
- No material factors were identified from the data collection that would adversely affect use of the data in Mineral Resource estimation.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

This section has not been changed from the Kipushi 2019 Resource Update and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Kipushi 2019 Resource Update.

11.1 Gécamines Sampling Approach

Sampling by La Générale des Carrières et des Mines (Gécamines) was selective and lower-grade portions of the mineralised intersections were not always sampled. Drill cores had a diameter of between 30–70 mm. The core sampling and sample preparation procedures were reported as follows:

- The drill cores were sawn in half.
- Sample lengths were based on homogenous zones of mineralisation ranging from less than 1 m to greater than 10 m in length with an average length of 3.44 m and divided into three categories (copper–copper / zinc, zinc, and copper-lead-zinc) and sampled.
- Waste material was not sampled.
- Remaining half core was placed in core trays and stored.
- Aggregated half core samples were sent to the Gécamines laboratory for crushing, splitting, milling, and sieving.

11.2 Gécamines Sample Preparation and Analytical Approach

All of the historical assays on samples generated by Gécamines drilling at Kipushi are believed to have been carried out at the Gécamines mine laboratory at Kipushi. Mr M Robertson from The MSA Group (Pty) Ltd (MSA) inspected the laboratory on 21 February 2013. Gécamines laboratory staff at the time of the visit were reportedly involved with the processing of the historical samples and provided the following insight into sample preparation and analytical procedures as well as quality control (QC) procedures in place at the time (Figure 11.1):

- Samples were prepared using a belt-driven jaw crusher and two roller crushers to a nominal size of <5 mm.
- A split of the crushed material was then ground in a pulveriser (which has subsequently been removed from the laboratory) to 100% <100 mesh.
- Compressed air and brushes were used to clean equipment. It is not clear whether barren flush material was also used.
- Sample analysis was carried out by a four-acid digest and AAS finish, for copper, lead, zinc, arsenic, and iron. Results were reported in percentages. The laboratory then made composite samples of grouped categories, analysed these for germanium, cobalt, silver, cadmium, and rhenium, and reported results in ppm. No gold analyses were undertaken. The original GBC Avanta AAS instrument is still operational.
- Sulfur analysis was carried out by the 'classical' gravimetric method.

- Various Gécamines internal standards were used, with a standard read after every 6th routine sample. A blank was reportedly read at the beginning of each batch. Repeat readings were also carried out; The QC results were apparently not reported on the assay certificates and the data are therefore not available.
- As an additional QC measure, samples were also reportedly sent to the central Gécamines laboratory in Likasi for check analyses.
- It does not appear that samples were submitted for check analysis to laboratories external to Gécamines.

Figure 11.1 Sample Preparation and Wet Chemistry Analytical Laboratory at Kipushi

A) Belt-driven jaw and roller crusher

B) Site of pulveriser (now removed) against far wall

C) GBC Avanta AAS instrument reportedly used in the original analytical work from 1990-1993

D) Diluted standards currently in use at the Kipushi laboratory wall

MSA, 2013

11.3 KICO Sample Preparation Methods

All sample preparation, analyses and security measures were carried out under standard operating procedures set up by Kipushi Corporation SA (KICO) for the Kipushi Project. These procedures have been examined by the Qualified Person (QP) (Michael Robertson) and are in line with industry good practice.

For drillholes KPU001 to KPU051, sample lengths were a nominal 1 m, but adjusted to smaller intervals to honour mineralisation styles and lithological contacts. From hole KPU051 onwards, the nominal sample length was adjusted to 2 m for all zones with allowance for reduced sample lengths to honour mineralisation styles and lithological contacts; sample lengths within the copper-rich zones are typically 1 m or less. Following sample mark-up, the drill cores were cut longitudinally in half using a diamond saw. Half core samples were collected continuously through the identified mineralised zones.

Sample preparation was completed by staff from KICO and its affiliated companies at its own internal containerised laboratories at Kolwezi and Kamoa-Kakula (Figure 11.2 and Figure 11.3 respectively). Between 1 June and 31 December 2014, samples were prepared at the Kolwezi sample preparation laboratory by staff from the company's exploration division. After 1 January 2015, samples were prepared at Kamoa-Kakula by staff from that project. The QP, Mr M Robertson inspected both sample preparation facilities on 25 April 2013. Representative subsamples were air freighted to the Bureau Veritas Minerals (BVM) laboratory in Perth, Australia for analysis.

Samples were dried at between 100°C and 105°C and crushed to a nominal 70% passing 2 mm, using either a TM Engineering manufactured Terminator jaw crusher or a Rocklabs Boyd jaw crusher. Subsamples (800–1,000 g) were collected by riffle splitting and milled to 90% passing 75 µm using Labtech Essa LM2 mills. Crushers and pulverisers were flushed with barren quartz material and cleaned with compressed air between each sample.

Grain size monitoring tests were conducted on samples labelled as duplicates, which comprise about 5% of total samples, and the results recorded. A total of 400 g of dry material was used for the crushing test, 10 g of dry material was used for the dry pulverised test, and 10 g of wet material was used for the wet pulverised test.

Subsamples collected for assaying and witness samples comprise the following:

- Three 40 g samples for Democratic Republic of Congo (DRC) government agencies.
- A 140 g sample for assaying at BVM.
- A 40 g sample for portable XRF analyses.
- A 90 g sample for office archives.

Figure 11.2 Containerised Sample Preparation Facility at the Kolwezi Laboratory

MSA, 2013

Figure 11.3 Sample Preparation Facility at the Kamoa-Kakula Laboratory

MSA, 2013.

11.4 KICO Analytical Approach

The laboratory analytical approach and suite of elements to characterise the major and trace element geochemistry of the Big Zinc for the underground drilling programme were informed by the results of an 'orientation' exercise (Figure 11.4). This was carried out by taking 10 quarter core samples from different mineralisation styles from Gécamines drillholes which intersected the Fault Zone and Big Zinc.

The orientation samples were submitted to both BVM and Intertek Genalysis in Perth, Australia for analysis by sodium peroxide fusion (SPF) and ICP finish, high-grade and standard four acid digest and ICP finish, and gold by fire assay and AAS finish. The results of the orientation sampling exercise are described in Robertson (2013).

BVM was selected as the primary laboratory for the underground drilling programme. Representative pulverised subsamples from the underground drilling were submitted for the following elements and assay methods, based on the results of the orientation sampling:

- Zn, Cu, and S assays by SPF with an ICP-OES finish.
- Pb, Ag, As, Cd, Co, Ge, Re, Ni, Mo, V, and U assays by peroxide fusion with an ICP-MS finish.
- Ag and Hg by Aqua Regia digest and ICP-MS finish.
- Au, Pt, and Pd by 10 g (due to inherent high sulfur content of the samples) lead collection fire assay with an ICP-OES finish.

Four silver (Aqua Regia) assays were used below approximately 50 ppm and SPF assays were used above approximately 50 ppm.

BVM is accredited by The National Association of Testing Authorities (NATA) in Australia, to operate in accordance with ISO/IEC 17025 (Accreditation number: 15833).

Figure 11.4 Re-sampling of Gécamines Core for Assay Orientation Purposes

MSA, 2013

11.5 Quality Assurance and Quality Control

11.5.1 QA/QC Approach

A comprehensive chain of custody and a quality assurance and quality control (QA/QC) programme was maintained by KICO throughout the underground drilling campaign.

Input into the QA/QC programme and SOP was provided by MSA. The QA/QC programme was monitored by Dale Sketchley of Acuity Geoscience Ltd and reported on for the initial period 1 May 2014 to 1 September 2015 in Sketchley (2015a, b, and c) and subsequently for the full period 1 May 2014 to 31 March 2018 (Sketchley, 2018). The results presented below are largely sourced from these reports.

QA/QC work comprised shipping of samples for preparation and assaying, liaising with sample preparation and assay laboratories, reviewing sample preparation and assay monitoring statistics, and ensuring non-compliant analytical results were addressed. The QA/QC programme monitored:

- Sample preparation screen test data.
- Analytical data obtained from certified reference materials (CRM), blanks (BLK), and crushed duplicates (CRD).
- Internal laboratory pulverised replicates (LREP) for BVM.

Elements reviewed comprised Zn, Cu, Pb, Ag, Au, Ge, S, As, Cd, Co, Hg, Re, Ni, Mo, V, and U. Elements with incomplete data that are mostly below or near the reported lower detection limits are not discussed further; these comprise Ni, Mo, V, U, Pt, and Pd.

All KICO data from the project is stored in an MS Access database. QA/QC data were exported from the Access database into software applications for creating monitoring charts and comparison charts. The number of samples reviewed by Sketchley (2018) comprised 12,944 routine samples, 655 CRMs, 599 blank samples, 450 crushed duplicates and 1,109 laboratory duplicates.

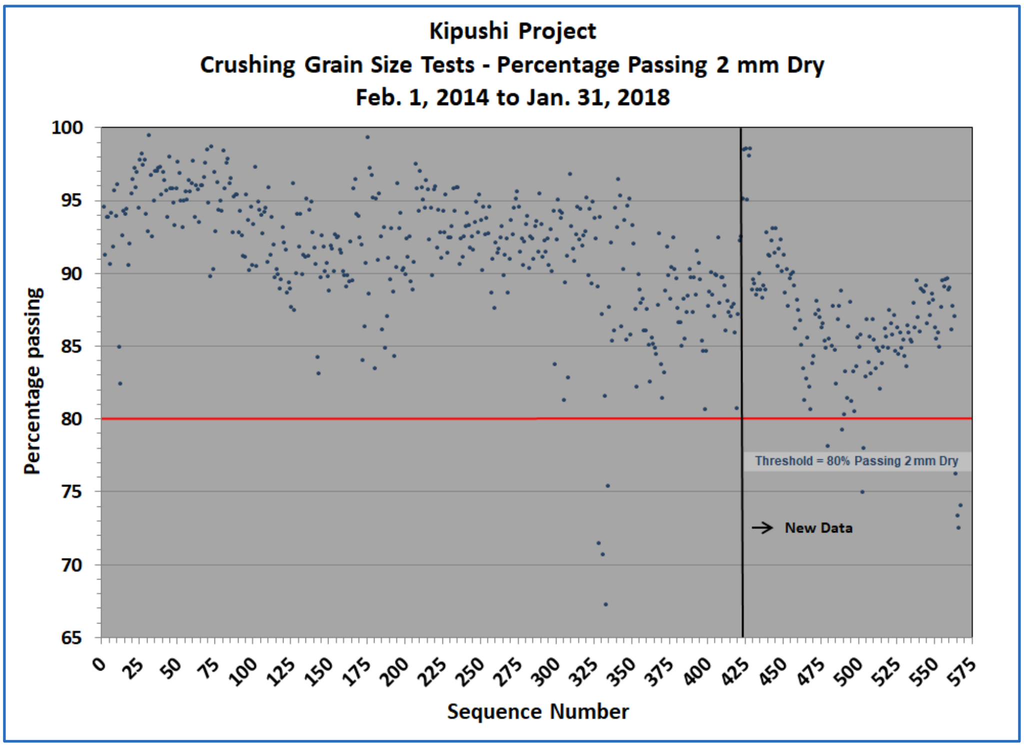
All of the sample batches submitted to BVM had approximately 5% CRMs, 5% blanks, and 5% crushed reject duplicates inserted into the sample stream.

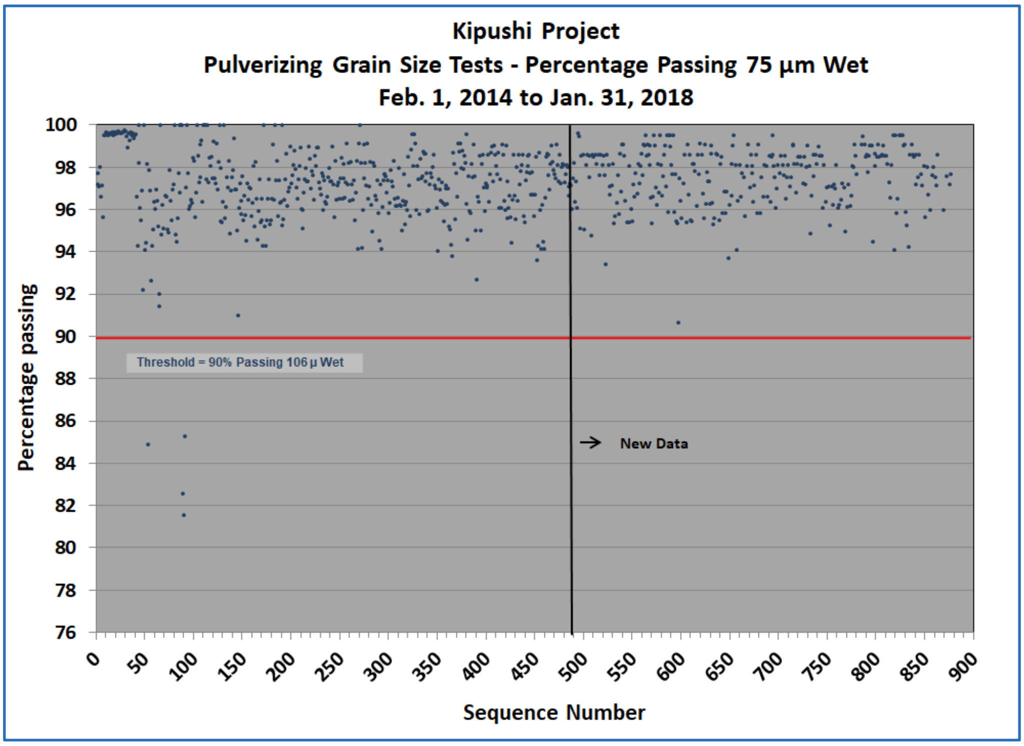
11.5.2 Laboratory Performance

11.5.2.1 Sample Preparation

Final statistical charts illustrating results from the Kolwezi and Kamoa-Kakula sample preparation laboratories grain size monitoring are presented in Figure 11.5. The majority of samples pass 80% dry for the crushing step. For the pulverising step, almost all samples pass 90% wet and the majority of samples pass 80% dry. The results are acceptable for styles of mineralisation with low heterogeneity.

Figure 11.5 Crushing and Pulverising Grain Size Monitoring Charts





Sketchley, 2018

11.5.2.2 Certified Reference Materials

CRMs were sourced from a number of independent commercial companies:

- Ore Research and Exploration (OREAS series) in Australia.
- Natural Resources Canada – Canadian Certified Reference Material Project (CCRMP series).
- African Mineral Standards (AMIS series), a division of Set Point Technology in South Africa.
- Matrix-matched CRMs from Kipushi processed by CDN Resource Laboratories Ltd (KIP series).

The AMIS, CCRMP, and OREAS series were used up to early 2015, and the KIP series preferentially thereafter. As the KIP series of CRMs was introduced late in the drilling programme, the results are of limited applicability for the entire data set. The CRMs were used to monitor the accuracy of laboratory assay results. Certified mean values and tolerance limits derived from a multi-laboratory round robin programme have been provided by the manufacturers and were used in the CRM monitoring charts. The CRMs used in the programme, together with the certified element concentrations, are listed in Table 11.1 and Table 11.2 respectively. These CRMs generally cover the observed grade ranges for Zn, Cu, Pb, Ag, S, Ge, Au, As, and Cd at Kipushi.

Analytical performance of the CRMs was monitored on an ongoing basis by KICO personnel using two to three standard deviation tolerance limits. The results of the CRM programme for the main elements of economic interest are shown in Table 11.3. Summary charts for zinc and copper are shown in Figure 11.6.

CRM assays were reviewed using sequential monitoring charts for Zn, Cu, Pb, Ag, Ge, Au, S, Cd, Co, Hg, and Re, annotated with the certified mean values, two and three standard deviations (2–3SD), and 5–10% tolerance limits. AMIS 83, AMIS 84, AMIS 144, and AMIS 149 were excluded from the QA/QC review as they were used only once each in the initial drilling programme.

CRM failures were defined as samples which returned assay results outside of the three standard deviation tolerance limits. In most cases, CRM failures were re-assayed together with several samples on either side, within the sample stream. In cases where CRM failures were not re-assayed, the adjacent routine samples were checked for elevated grades in order to assess the impact.

CRM performance was assessed for data above the following thresholds: Zn >1%, Cu >1%, Pb >1%, Ag (Aqua Regia) >11 ppm and <50 ppm, Ag (SPF) >50 ppm, Ge >10 ppm, Au >25 ppb, all S, As >500 ppm, Cd >500 ppm, Co >500 ppm, Hg >0.1 ppm, and Re >0.1 ppm. These thresholds were used to eliminate lower value data well below economic cut-off grades and closer to the lower detection limits where analytical performance is typically poor, especially for the SPF method.

Table 11.1 Commercial CRMs Used in the KICO Drilling Programme

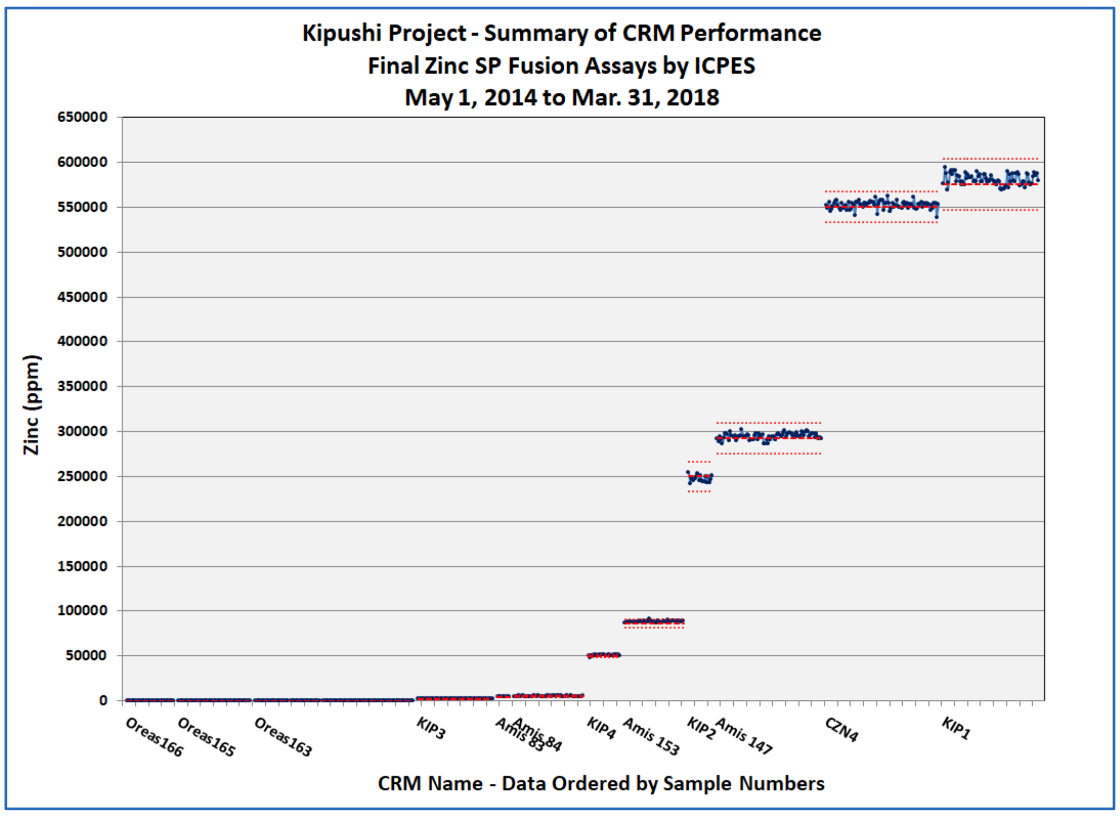
CRM	Commodity	Minerals	Source	Geological Setting	Location
AMIS 83	Zn, Pb, Cu, Ag	Sp, Gn + Zn-Pb Oxides	Kihabe – Nxuu Project	Neo-Proterozoic SEDEX deposit	Botswana
AMIS 84	Zn, Pb, Cu, Ag	Sp, Gn + Zn-Pb Oxides	Kihabe – Nxuu Project	Neo-Proterozoic SEDEX deposit	Botswana
AMIS 144	Zn, Cu	Zn Oxides	Skorpion Mine	Proterozoic SEDEX deposit	Namibia
AMIS 147	Zn, Ag, Cu, Pb	Sp, Gn, Py, Cp	Rosh Pinah Mine	Proterozoic SEDEX deposit	Namibia
AMIS 149	Zn, Ag, Cu, Pb	Sp, Gn, Py, Cp	Rosh Pinah Mine	Proterozoic SEDEX deposit	Namibia
AMIS 153	Zn, Ag, Cu, Pb	Sp, Gn, Py, Cp	Rosh Pinah Mine	Proterozoic SEDEX deposit	Namibia
CZN4	Zn, Ag, Cu, Pb	Sp, Py, Po, Cp	Kidd Creek Mine	Archaean VMS deposit	Canada
Oreas 163	Cu	Cp, Py, Po	Mt. Isa Mine	Mid-Proterozoic dolomitic shale	Australia
Oreas 165	Cu	Cp, Py, Po	Mt. Isa Mine	Mid-Proterozoic dolomitic shale	Australia
Oreas 166	Cu	Cp, Py, Po	Mt. Isa Mine	Mid-Proterozoic dolomitic shale	Australia
Kip 1	Zn, Cu, Pb, Ag, Ge, Au	Sp, Cp, Py, Bn, Gn	Kipushi Mine	Proterozoic Central African Copperbelt	DRC
Kip 2	Zn, Cu, Pb, Ag, Ge, Au	Sp, Cp, Py, Bn, Gn	Kipushi Mine	Proterozoic Central African Copperbelt	DRC
Kip 3	Zn, Cu, Pb, Ag, Ge, Au	Sp, Cp, Py, Bn, Gn	Kipushi Mine	Proterozoic Central African Copperbelt	DRC
Kip 4	Zn, Cu, Pb, Ag, Ge, Au	Sp, Cp, Py, Bn, Gn	Kipushi Mine	Proterozoic Central African Copperbelt	DRC

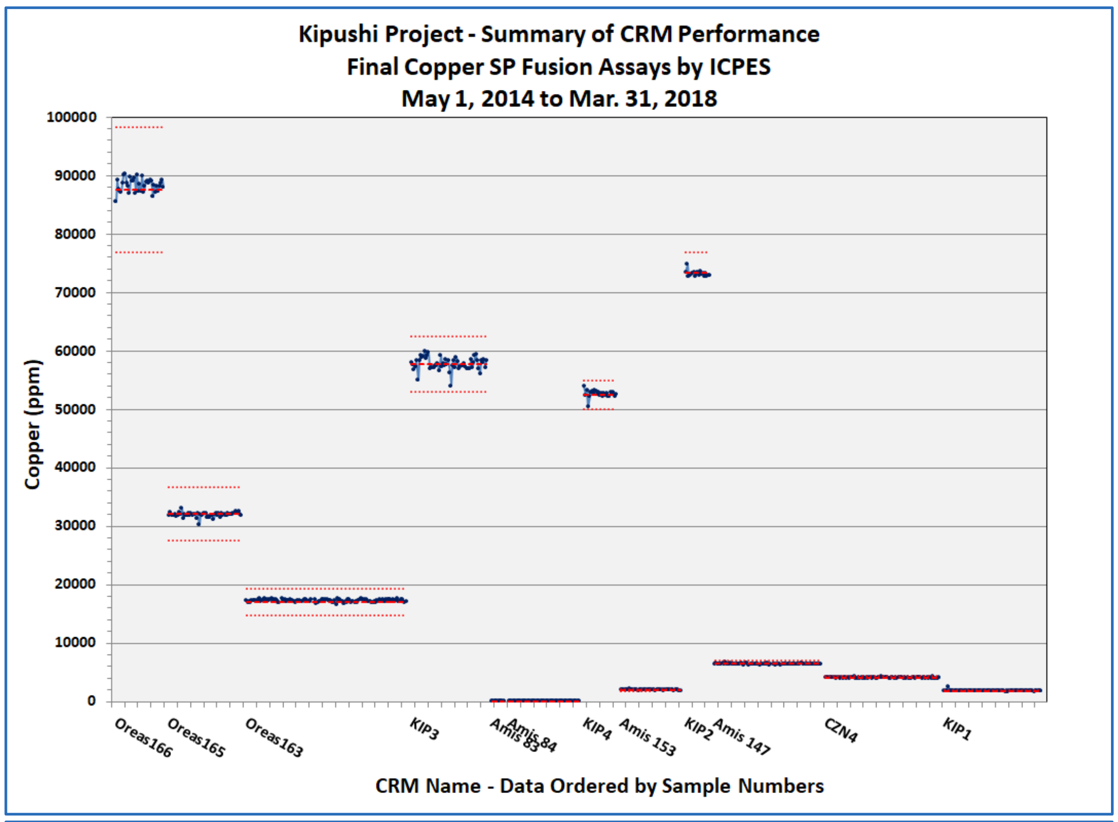
Table 11.2 Certified Concentrations by Sodium Peroxide Fusion for CRMs used in the KICO Drilling Programme

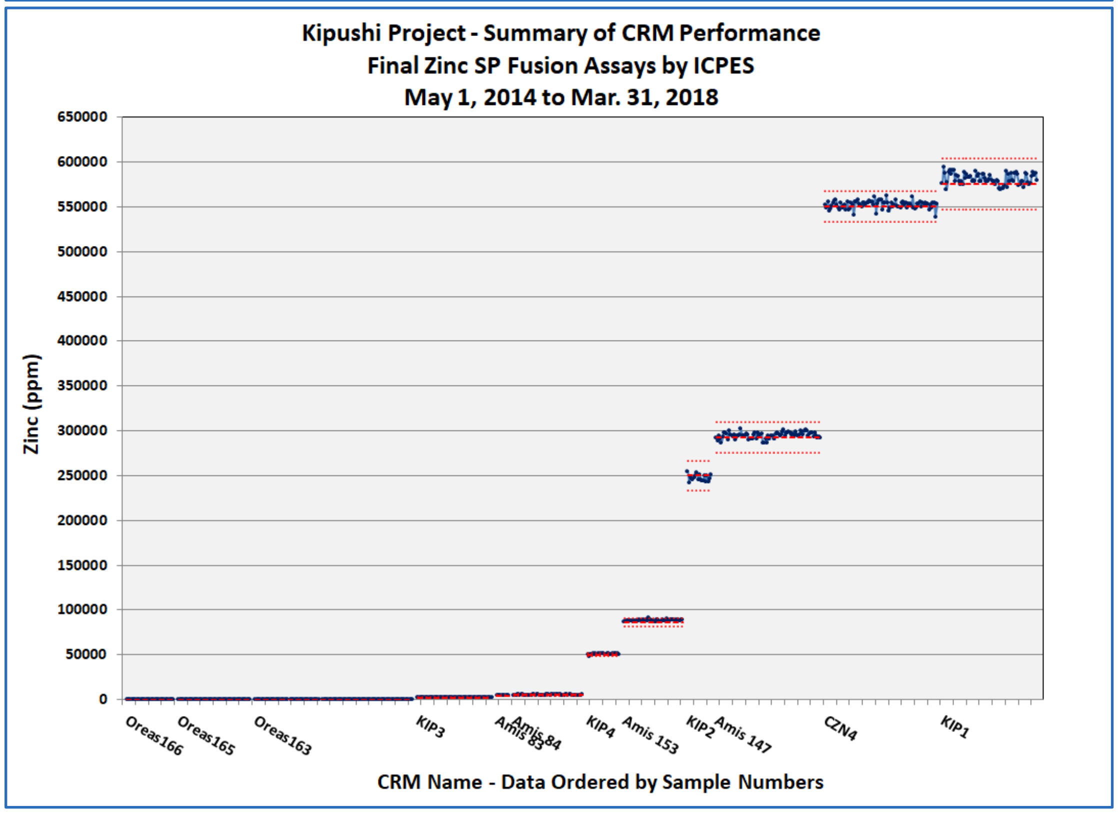
CRM	Zn (%)	Cu (%)	Pb (%)	Ag (AR ppm)	Ag (ppm)	Ge (ppm)	Au (FA) (ppb)	S (%)	As (ppm)	Cd (ppm)	Co (ppm)	Hg (ppm)	Re (ppm)
AMIS 83	–	–	–	–	–	–	–	–	–	–	–	–	–
AMIS 84	–	–	–	–	–	–	–	20.06	–	–	–	–	–
AMIS 144	–	–	–	–	–	–	–	–	–	–	–	–	–
AMIS 147	29.05	–	3.32	–	62.8	–	360	–	–	647	–	–	–
AMIS 149	–	–	–	–	–	–	–	–	–	–	–	–	–
AMIS 153	8.66	–	1.02	19.90	–	–	230	6.00	–	–	–	–	–
CZN4	55.07	–	–	–	51.4	–	–	33.07	–	2,604	–	4.54	–
Oreas 163	–	1.71	–	–	–	–	–	9.98	–	–	–	–	–
Oreas 165	–	10.20	–	–	–	–	–	8.28	–	–	2,485	–	–
Oreas 166	–	8.75	–	10.80	–	–	–	11.29	–	–	2,077	–	–
Kip 1	57.57	–	–	21.20	–	88.0	26	34.06	908	3,254	–	–	–
Kip 2	25.01	–	–	–	165.0	49.3	96	24.07	1,401	1,548	–	–	0.188
Kip 3	–	5.78	–	36.00	–	–	–	6.10	1,431	–	–	–	0.875
Kip 4	5.00	5.24	–	22.20	–	11.5	51	17.00	2,327	–	–	–	–

AR = Aqua Regia; FA = Fire Assay.

Figure 11.6 CRM Performance Summary for Zinc and Copper







Sketchly, 2018

Table 11.3 CRM Performance for the Main Elements of Economic Interest

Element	Accuracy and Precision	Failures
Zn	Mean values within 2% of the certified values and RSD values <2%.	CZN4 and AMIS 147 each had one positive failure. Re-assays addressed the CZN4 failure, whereas the one for AMIS 147 remains and is most likely due to a mix-up with a routine sample as the multi-element signature does not match any of the CRMs.
Cu	Mean values within 2% of the certified values and RSD values <2%.	Oreas 165 and 166 each had one failure, which was due to misclassification. The database was corrected to address the issue.
Pb	Mean values within 1% of the certified values and RSD values <3%.	AMIS 147 had four positive failures, and AMIS 153 had three positive failures. Three of the four failures for AMIS 147 and two of the three for AMIS 153 were re-assayed with surrounding samples, which addressed the failures. One positive failure for AMIS 147 remains and is most likely due to a mix-up with a routine sample as the multi-element signature does not match any of the CRMs. The sample data were removed from the statistical summary. One marginal positive failure for AMIS 153 remains, which has negligible impact.
Ag (AR)	Accuracy and precision for all CRMs is poor. Mean values are negatively biased up to 9%, and most RSD values are in the range 7–9%.	A number of failures (mostly negative) were observed. No failures were re-assayed due to the overall negative bias, which will also apply to the routine sample Ag values. Values above 50 ppm are outside the acceptable range for the method, with the strong negative bias due to the partial digest of the method. Due to the classification of Kip 2 as Provisional, there are no tolerance limits to classify samples as passed or failed.
Ag (SFP)	Accuracy and precision for the AMIS and CZN CRMs is poor. AMIS 147 displays a negative bias of 6% and a RSD of 8%. CZN4 shows a negative bias of <2% and a RSD of 9%.	A number of negative failures remain for AMIS 147, with one likely due to a sample mix-up as the multi-element signature does not match any the CRMs. Re-assays returned values well below the range of the method for the surrounding routine samples; therefore, the impact of the failures is regarded as negligible. CZN4 displays multiple negative failures due to poor resolution of the method.
Ge	Accuracy and precision for all three CRMs is poor.	KIP 1 displays no failures despite a strong negative bias of almost 11% in the first sampling campaign and no negative bias in the 2018 data. The single KIP 2 result is a marginal negative failure in phase 1 and has three failures, related to a positive bias and wider tolerance limits from the RR certification programme. KIP 4 displays one positive failure and poor precision, in each phase, due to the low value.
Au (FA)	Accuracy and precision for all CRMs tends to be poor.	AMIS 147 displays two marginal positive failures and a negative failure likely due to sample mix-up. AMIS 153 displays a negative bias of 12% although no failures. The remaining CRMs have low gold values and the impact of failures is regarded as negligible. Gold assays were discounted during the 2018 drilling programme.

Element	Accuracy and Precision	Failures
S	Accuracy and precision for all CRMs is good with mean values within 2% of the certified values and RSD values <3%.	CZN4 has one marginal positive failure remaining, which has a minor impact. Oreas 165 and 166 each had one failure, which was due to misclassification. The database was corrected to address the issue.

AR = Aqua Regia; SFP = Sodium Peroxide Fusion; FA = Fire Assay.

11.5.2.3 Blanks

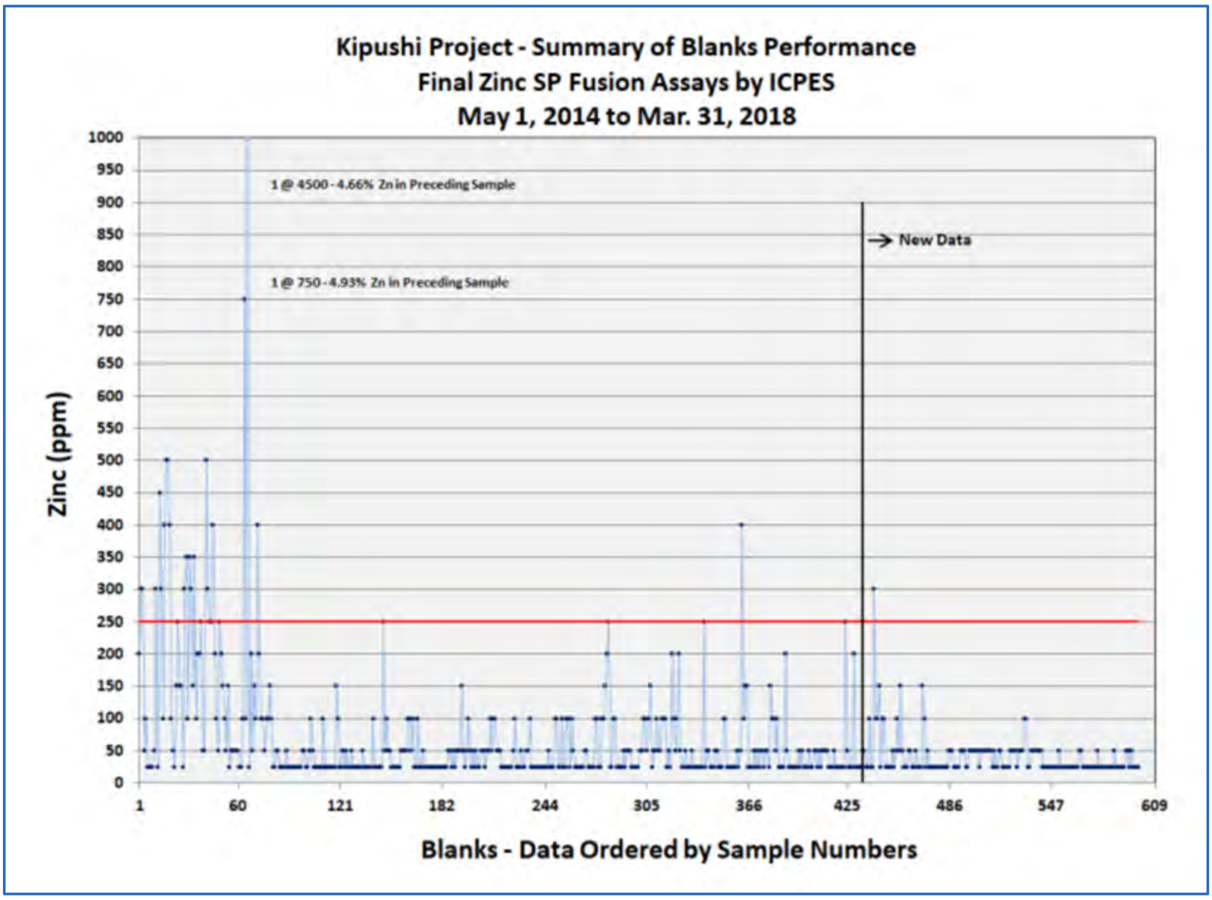
Locally obtained barren coarse quartz vein material was used to monitor contamination and sample mix-ups (Figure 11.2). This material was previously analysed in separate programmes (both Kipushi re-sampling and Kamoa-Kakula programmes) to ensure that it was barren of the elements of interest. Analytical performance of blank samples was evaluated on an ongoing basis by KICO personnel using threshold limits. Where failures over thresholds were identified, the blank and a group of adjacent samples were submitted for re-assaying of the failed elements. Re-assays were evaluated in the same manner. The results suggest a low level of Zn contamination during pulverising.

Blank sample assays were monitored using sequential control charts for Zn, Cu, Pb, Ag (Aqua Regia), Ag (peroxide fusion), Ge, Au, S, As, Cd, Hg, and Re and annotated with threshold limits.

Blank sample monitoring results for zinc by SPF are shown in Figure 11.7. A large number of failures are observed at the beginning of the programme. These are related to a combination of four causes: sample bags damaged in shipment to BVM; cleaning material submitted for assaying instead of actual blank material; carry-over from extremely high-grade samples; and zinc in pulverising bowl material. The first two were rectified, leaving the remaining failures related to carry-over from preceding samples and pulverising bowl material. Most of the failures are in the range of several hundred ppm and are well below economic cut-off values, however, one failure is quite high at 4,450 ppm, and it was re-assayed together with surrounding samples in the sequence. The re-assays confirmed the higher value, which is most likely related to the carry-over from the preceding higher-grade sample. As the single sample is well below economic cut-off grade, it would have a negligible impact on any estimate.

The remaining elements have a small number of individual failures that are mostly lower values, except for one sample for gold at 835 ppb. The sample with high gold was repeated three times by BVM and returned between 663 ppb and 2,000 ppb. The anomalous values may be related to spurious gold within the quartz vein material.

Figure 11.7 Blank Sample Performance for Zinc by Sodium Peroxide Fusion



Sketchley, 2018

11.5.2.4 Duplicates

Crushed duplicate samples were obtained by riffle splitting of 2 mm crushed samples and were inserted into the sample stream to monitor the precision of the combined crushing and pulverising stages of sample preparation as well as the analytical stage. Most of the observed differences in duplicate pairs can generally be attributed to splitting at the crushing stage.

Pulverised duplicates were routinely done by BVM during assaying and were used to monitor the combined precision of the pulverising stage of sample preparation and the analytical stage.

Bias was evaluated using Scatter, Quantile, and Relative Difference plots, with precision guidelines at ±10%, 20%, and 30%. Patterns for most elements are symmetrical about parity, thereby suggesting no biases in the sample preparation and assaying process. Reduced major axis (RMA) equations indicate biases are less than 1% for most elements. Exceptions are silver (Aqua Regia), silver (peroxide fusion), gold, and rhenium. Silver (Aqua Regia) has an increase in scatter above 50 ppm, which is the upper limit of the method. The bias decreases to near 1% when data above this threshold are excluded, although the original samples tend to have a slight negative bias. Silver (peroxide fusion) has an increase in scatter for data above 125 ppm. The bias decreases to near 1% when data above this threshold are excluded.

Both gold and rhenium have a greater degree of scatter for all grades and noticeable differences in values for several sample pairs where the duplicate is significantly lower than the original. The bias decreases to near 1% when these data are excluded.

Precision was evaluated using Absolute Relative Difference by grade, Absolute Relative Difference by percentile and Thompson Howarth plots. Precision levels using global Absolute Relative Difference by grade for crushed duplicates are 4–13% for all elements except gold and rhenium, which are 42% and 23% respectively. Differences for pulverised duplicates are 4–11% for all elements except gold, which is 33%.

Precision levels using Absolute Relative Difference by Percentile were compared to maximum ideal differences at the 90th percentile of 20% for crushed duplicates (CRDs) and 10% for laboratory repeats (LREPs). Copper, silver (Aqua Regia), sulfur and cadmium all have absolute relative differences at or less than the maximum ideal thresholds of 20% for CRDs and 10% for LREPs. Larger differences for zinc, lead, silver (Peroxide Fusion), germanium, gold, arsenic, cobalt, and mercury are related to large numbers of lower value data with poor repeatability. When the data below five to ten times the lower detection limit are excluded, most of the differences decrease to less than 20% for CRDs and 10% for LREPs. Larger differences for silver (peroxide fusion), gold and cobalt are related to a greater degree of scatter for all grades.

Precision using the Thompson Howarth method was evaluated utilising the level of Asymptotic Precision and the Practical Detection Limit. Asymptotic Precision is defined as the level of variability at values well above the lower detection limit. Practical detection limit is the grade where the level of precision equals 100% and indicates data are completely random below this threshold. As a general guideline, depending on actual heterogeneity, the asymptotic precision should be better than 10–20% for crushed duplicates, and better than 5–10% for pulverised duplicates.

Asymptotic precision values for CRDs and LREPs are 10% or below for all elements, except gold, which have a level of 19% for CRDs and 13% for LREPs. All elements tend to have better precision for pulverised duplicates than crushed duplicates, as expected. Similarly, the practical detection limit for pulverised duplicates tends to be better than for crushed duplicates and higher than the actual instrumental lower detection limits.

11.5.2.5 Second Laboratory Check Assay Programme

An initial check assay programme was undertaken on a set of representative samples from drillholes KPU001 to KPU025, in order to confirm the assays from the primary laboratory BVM. This work is reported on in Sketchley (2015b). A subsequent check assay programme was carried out on samples from drillholes KPU026 to KPU072 and reported in Sketchley (2015c).

The check samples were selected on a random basis, representing 10% of the total sample population after excluding all samples that reported less than 0.1% Zn and 0.1% Cu. The selection was supplemented by additional samples that reported higher Ge, Re, and mixed Zn/Cu, in order to round out the grade profile for the final set of samples for check assaying.

Sample material was sourced from archived pulps (i.e. not the reject pulps from the BVM assays) prepared and stored at the Kolwezi sample preparation facility. The sample batch submission also contained an appropriate quantity of CRMs, pulp blanks and duplicates. CRMs that were routinely used for the project submissions to BVM were used for quality control in the check assay batches. Duplicate check sample batches were submitted to the Intertek Genalysis (Intertek) and SGS laboratories in Perth (SGS). Analytical methods were matched as closely as possible to those used by the primary laboratory, BVM.

The quality of the check assay results was assessed using sequential CRM and blank sample monitoring charts and scatterplots for duplicate pairs. Failures were subjected to re-assay including several samples from the sequence on either side of the failed assay.

In the initial check assay programme, failures for higher-grade Zn, Cu, Pb, Ag, and S CRMs assayed by SGS were more frequent than for Intertek. The Intertek results show a slight overall negative bias for most elements, whereas SGS results show a slight overall positive bias for most elements. Although both laboratories validated the original assays conducted by BVM, the Intertek results were more stable than SGS, with fewer issues, and Intertek was selected for all subsequent check assay work.

Intertek generally performed well based on the Kipushi matrix-matched CRMs used in the latter part of the programme. CRM failures are generally related to lower values well below economic cut-offs.

11.5.3 Conclusions

The QA/QC protocol implemented by KICO concluded the following:

- The results of the QA/QC programme demonstrate that the quality of the assay data for zinc, copper, and lead is acceptable for supporting the estimation of Mineral Resources. Higher-grade assays for silver, germanium, and gold are useable, but the limitations in the quality of the data should be taken into account.
- The second laboratory check assay programme conducted by Intertek validated the original BVM assays for most elements. Any future checking work should continue to use the Intertek laboratory; however, issues with carry-over need to be re-emphasised.

- Sample material for the second laboratory check assay programme was sourced from archived pulps (i.e. not the same pulps assayed by BVM) stored at the Kolwezi sample preparation facility. Future check assays should be conducted on the assay pulp residues remaining from the BVM assays.
- Gécamines did not carry out routine check assaying. Check assays were only carried out when visual grade estimates did not correspond with the laboratory results. Gécamines protocol for internal check sampling is unknown and there was no check assaying or sampling by an independent external laboratory.
- No data are available for QA/QC routines implemented for the Gécamines samples, therefore, the Gécamines sample assays should be considered less reliable than the KICO sample assays.

11.6 Security of Samples

Historically the sample chain of custody is expected to have been good as the samples did not leave the site and were assayed at the Gécamines laboratory at Kipushi. The split mineralised core material was retained on site in a core storage building. The rejects and pulps were also stored, but over the years many were destroyed or lost.

KICO maintains a comprehensive chain of custody programme for its drill core samples from Kipushi. All diamond drill core samples are processed at either the company's Kolwezi facility, or at the Kamoa-Kakula Project facility. Core samples are delivered from Kipushi to the sample preparation facility by company vehicle. On arrival at the sample preparation facility, samples are checked, and the sample dispatch forms signed. Prepared samples are shipped to the analytical laboratory in sealed sacks that are accompanied by appropriate paperwork, including the original sample preparation request numbers and chain-of-custody forms.

Paper records are kept for all assay and QA/QC data, geological logging and specific gravity information, and downhole and collar coordinate surveys.

12 DATA VERIFICATION

This section has not been changed from the Kipushi 2019 Resource Update and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Kipushi 2019 Resource Update.

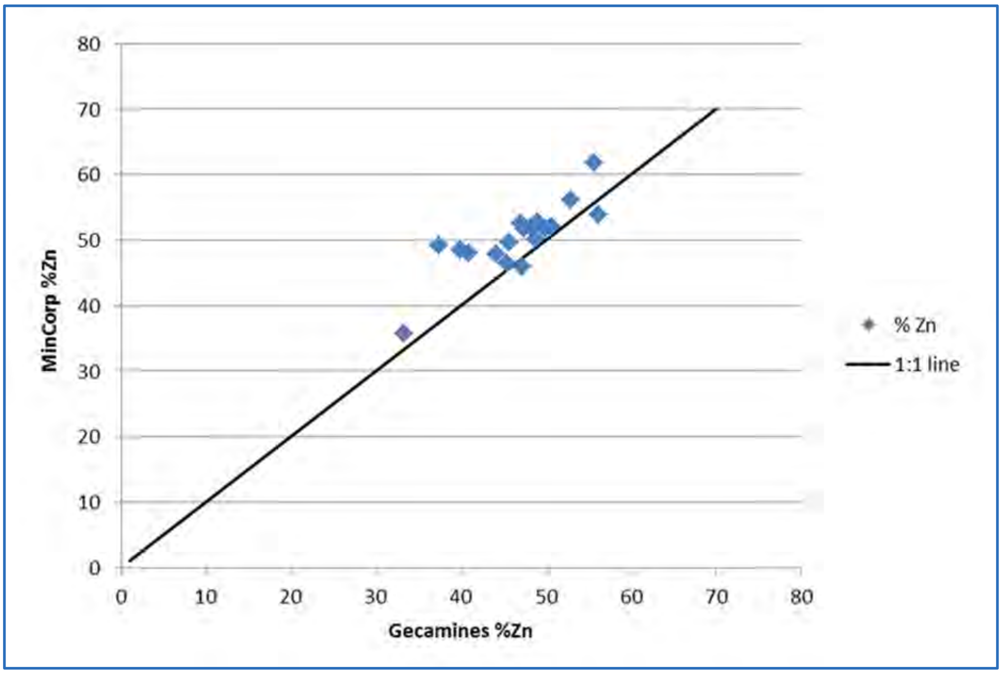
A comprehensive re-sampling programme was undertaken on historical La Générale des Carrières et des Mines (Gécamines) drillhole core from the Big Zinc Zone and Fault Zone below 1,270 mL at the Kipushi Mine. The objective of the exercise was to verify historical assay results and to assess confidence in the historical assay database for its use in Mineral Resource estimation.

In addition, Kipushi Corporation SA (KICO) completed a number of twin holes on the Big Zinc Zone between March 2014 and May 2015 with the objective of verifying historical Gécamines results.

12.1 Previous Re-sampling Programme (Mineral Corporation)

A limited re-sampling exercise was carried out by The Mineral Corporation that collected 20 x 2 m samples from 14 holes that intersected the Big Zinc Zone. These were analysed by Golden Pond Tr 67 (Pty) Ltd in Johannesburg using a 'full acid digest' and ICP finish. With the exception of two samples, all reported slightly higher results compared to the original Gécamines data (Figure 12.1). On the basis of this small population it was found that the Gécamines results under-report zinc by approximately 8% compared to the check assays.

Figure 12.1 Comparison between Gécamines and Mineral Corporation Zinc Assays on the same Sample Intervals



MSA, 2014

12.2 Big Zinc and Fault Zone Re-sampling Programme

12.2.1 Sample Selection

An initial site visit to Kipushi was undertaken from 20–22 February 2013 by the Qualified Person (QP), Mr Robertson, in order to view the condition of the existing Gécamines drillhole cores from holes collared on the 1,270 mL, as well as to review existing hard copy plans, sections, drillhole logs, and assay results. The Gécamines laboratory at Kipushi was inspected and the staff were interviewed in order to establish the procedures used in the original preparation and analysis of the Kipushi drill core samples.

The availability of holes for the re-sampling campaign was constrained by the following factors:

- Drill cores are preserved from only 49 out of 60 holes.
- Limited re-sampling of 14 of the 49 holes was carried out by The Mineral Corporation resulting in only quarter core remaining in places.
- Core recovery issues in some holes.
- Some holes only had composite assay data results and individual sample assays were not available or were not captured.

Holes were selected to cover the various mineralisation styles and intervening low-grade 'sterile' zones (where core is preserved) and to cover the extent of the deposit. One hole was selected from each of the eight sections in order to cover the strike extent of the Big Zinc Zone and to allow for re-sampling of the Fault Zone where possible. The selected drillhole inclinations ranged from –25° to –75° to cover the dip extent of the mineralisation. The selected holes are listed in Table 12.1. These holes comprise 161 original sample intervals which represent approximately 16% of the historical sample database for the Big Zinc.

Re-sampling of the drill core was supervised by the The MSA Group (Pty) Ltd (MSA) QP in a follow-up site visit from 22–24 April 2013. Re-sampling was carried out using an average sample length of 1.9 m, compared to the original average sample length of 3.8 m (while honouring the original sample boundaries), in order to obtain better resolution on grade distribution. Direct comparison with the original sample lengths was subsequently carried out on a length weighted average grade basis.

Table 12.1 Holes Selected for Re-sampling

Level	Section	Resampling by MinCorp	Selected Hole	No. Original Samples	Comment
1,270	3	–55, –75	–75	31	Medium Cu zone in Fault Zone; wide intersection though Big Zinc Zone, although not true thickness
1,270	5	–55, –65, –75	–30	22	Intersects upper part of Big Zinc Zone, exhibits lower-grades. Two high Cu zones in Fault Zone. Individual assays available and need to be captured.
1,270	7	–55, –75	–25	21	Thick high Cu zone in Fault Zone; intersects upper part of Big Zinc Zone
1,270	9	–40, –75	–40	25	Medium Cu zone on Fault Zone; intersects entire middle zone of Big Zinc Zone; (–85 hole core not available therefore not an option)
1,270	11	–45, –65	–25	15	Intersects upper part of Big Zinc Zone; includes narrow zones of high Cu
1,270	13	–65	–75	19	Narrow zones of high Cu; intersects lower part of Big Zinc Zone
1,270	15	–20	–40	12	High Cu in Fault Zone; intersects middle zone of Big Zinc Zone
1,270	17	–70	–75*	16	Intersects lower part of Big Zinc Zone

* Core trays labelled –70.

12.2.2 Sample Preparation and Assay

A total of 384 quarter core samples (NQ size core) were collected and submitted to the KICO affiliated containerised sample preparation laboratory in Kolwezi for sample preparation. This facility and the sample preparation procedures were inspected by the QP on 24 April 2013 and found to be suitable for preparation of the Kipushi samples.

A total of 457 samples including quality control (QC) samples were submitted to the Bureau Veritas Minerals (BVM) laboratory in Perth, Australia for analysis by a combination of methods as shown in Table 12.2. Density determinations on every tenth sample were carried out at BVM using the gas pycnometry method.

Check (second laboratory) analyses of Zn, Cu, Pb, Ge, and Ag were carried out at the Perth-based Intertek Genalysis laboratories (Intertek) using the same assay methodology apart from Ag which was determined by four-acid digest and ICP-MS finish.

Table 12.2 Assay Methodology Approach

Method and Code	Elements
Fire Assay – ICP-AES finish (Doc 600)	Au, Pt, Pd
SPF with ICP-AES finish (Doc 300)	Ag, As, Cu, Fe, Pb, S, Zn
SPF with ICP-MS finish (Doc 300)	Ag, As, Ba, Be, Bi, Cd, Ce, Cs, Co, Cu, Dy, Er, Eu, Ga, Gd, Ge, Hf, Ho, In, La, Li, Lu, Mo, Nb, Nd, Ni, Pb, Pr, Rb, Re, Sc, Sm, Sn, Sr, Ta, Tb, Th, Tl, Tm, U, W, Y, Yb, Zr
Mini Aqua Regia digest with ICP-MS finish (Doc 403)	Hg

12.2.3 Assay Results and QA/QC

Quality control samples inserted into the sample stream comprised 16 coarse silica blanks, 18 coarse crush field duplicates and 40 standard samples from 15 certified reference materials (CRMs). The CRMs were selected to cover the grade range for Zn (0.30–55.24% Zn) and are certified for a variety of Cu, Pb, S, Ag, Fe, As, Cd, and Co.

CRM over-reporting failures for Zn and S were observed in the initial BVM assays, which led to a re-assay of Zn and S for all 457 samples. The over-reporting was confirmed by the results of 128 pulp splits analysed at a second laboratory (Intertek Genalysis in Perth). Although an improvement in the accuracy of results was noted in the re-assays, CRM failures for Zn and S were still observed and this was brought to the attention of BVM who re-analysed 120 samples for Zn and S using a modified approach. These results were regarded by the QP as acceptable. BVM was then requested to re-analyse all 457 samples for Zn and S in order to provide a 'clean' set of data. These final re-assays, together with the other multi-element results, which were accepted from the initial BVM work, comprise the final assay dataset for the re-sampling programme. A comparison of mineralised intersections, at a cut-off of 7% Zn, between historical and re-sampling results is shown in Table 12.3. The comparison revealed under-reporting by Gécamines for grades above 25% Zn, and over-reporting at grades less than 20% Zn (Figure 12.2). Several outlier pairs were observed that are likely to result from mixed core or discrepancies in depth intervals. This can be expected considering that the original drilling, sampling, and assaying took place some 20 years ago. If the obvious outliers are143143ere143y143143, the BVM results are on average 5.5% higher than the Gécamines results. A general under-reporting by Gécamines was also concluded from earlier re-sampling of 20 sample intervals by Mineral Corporation.

The observed discrepancies may be in part be due to a difference in analytical approach, with the original assays having been carried out by Gécamines at the Kipushi laboratory by a four-acid digest and AAS finish, for Cu, Co, Zn, and Fe rather than the sodium peroxide fusion (SPF) used by BVM.

Results for the other elements of interest are as follows:

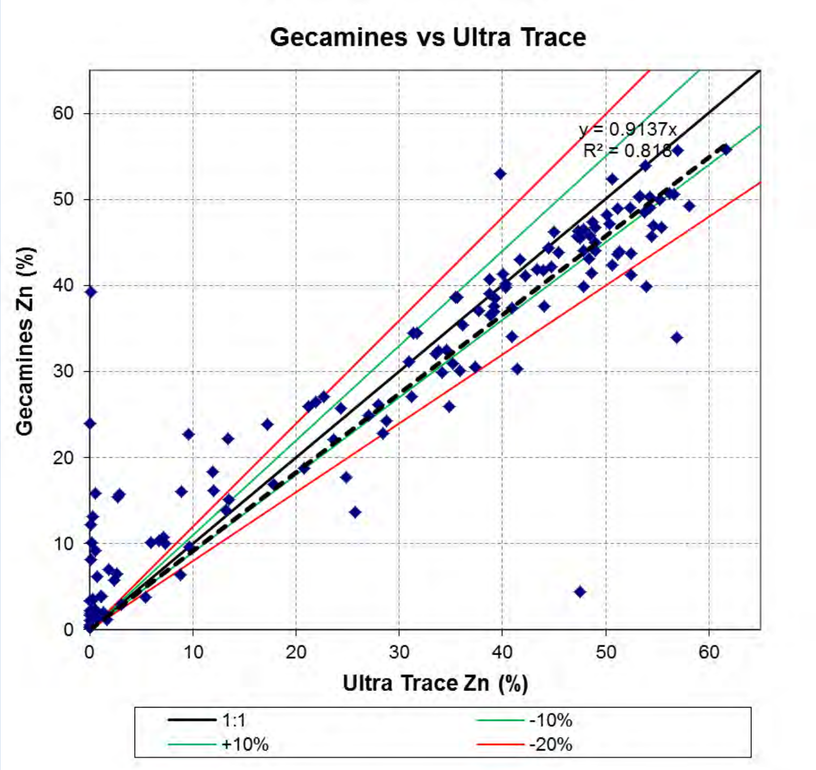
- Several outlier pairs are observed in the copper results that are likely to result from mixed core or discrepancies in depth intervals. Apart from the obvious outliers, a general correlation is observed between Gécamines and BVM that is considered acceptable, given the nuggety style of copper mineralisation.
- Disregarding the few outliers, BVM slightly under-reports lead compared to Gécamines.
- Sulfur displays a similar pattern to zinc, with slight over-reporting at higher-grades and under-reporting at lower-grades by BVM compared to Gécamines.
- Gold was not routinely reported in historical assays but was reported as part of the re-sampling programme. Grades are typically low with a maximum of 0.21 ppm gold reported.
- Germanium results are in line with historically reported results, although these were not reported routinely by Gécamines. The BVM germanium results are shown as a histogram plot in Figure 12.3.

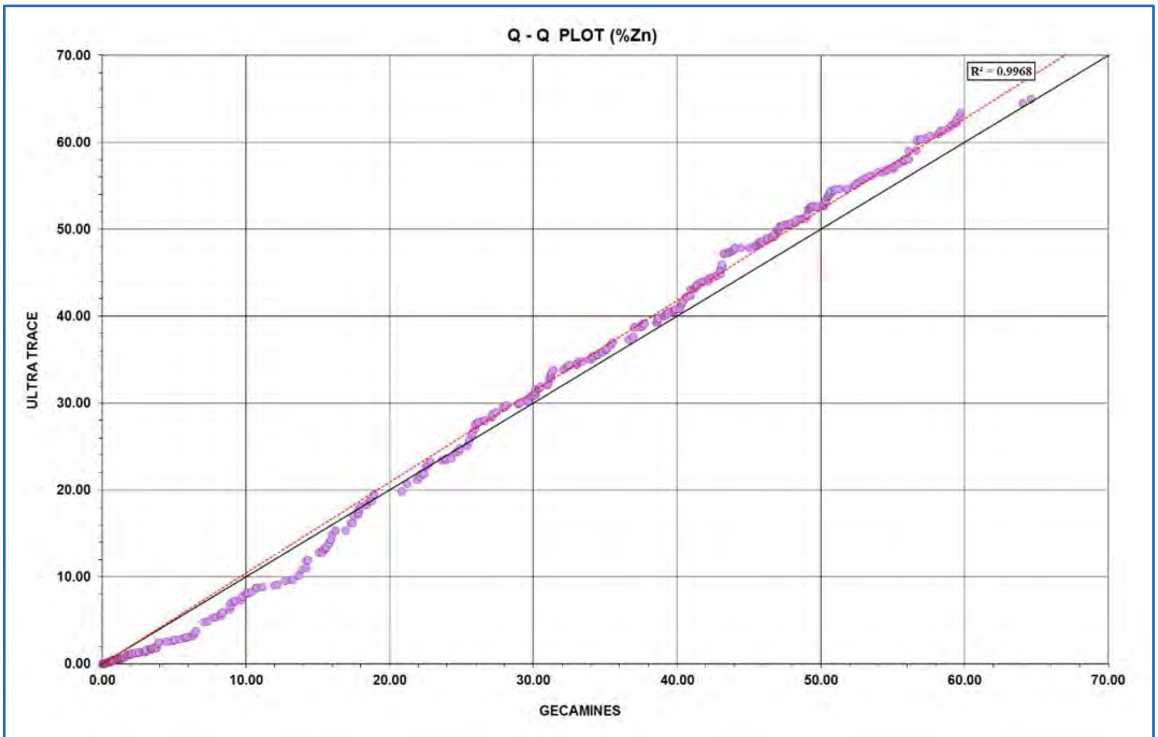
Table 12.3 Comparison of Mineralised Intersections between Gécamines and the Re-sampling Programme Using a Cut-off of 7% Zn

Hole_ID	Gécamines Data						Re-Sampling Programme					
	From	To	Interval[2]	Zn (%)	Cu (%)	Calculated Density	From	To	Interval[2]	Zn (%)	Cu (%)	Density[3]
1270/3/V+30/–75/SE1	99.00	219.30	120.30	36.11	0.69	3.50	124.80	303.70	178.90	48.01	0.28	4.09
1270/5/V+30/–30/SE	63.60	117.80	54.20	41.40	1.86	3.65	65.60	117.80	52.20	41.77	2.03	3.65
1270/5/V+30/–30/SE	142.50	155.60	13.10	18.74	0.97	3.21	153.75	155.60	13.10	20.76	0.45	3.75
1270/7/V+30/–25/SE	73.30	116.30	43.00	35.49	4.11	3.69	73.30	114.20	40.90	35.87	4.22	No data
1270/7/V+30/–25/SE	129.60	149.80	20.20	49.13	0.10	3.70	129.60	154.00	24.40	43.21	0.26	No data
1270/9/V+30/–40/SE	81.30	161.60	80.30	39.61	0.30	3.55	81.30	161.60	80.30	45.41	0.28	3.96
1270/11/V+30/–25/SE	72.50	123.50	51.00	21.78	1.16	3.27	82.90	123.50	40.60	20.28	0.42	3.44
1270/13/V+45/–75/SE	147.10	190.30	43.20	22.51	1.05	3.37	160.90	190.30	29.40	33.87	0.20	4.01
1270/15/W/–40/SE	90.10	98.20	8.10	29.03	0.48	3.44	90.10	98.20	8.10	29.03	0.45	3.99
1270/15/W/–40/SE	121.20	133.70	12.50	31.46	1.34	3.53	113.80	133.70	19.90	24.47	0.68	3.42
1270/17/W/–75/SE	127.80	135.10	7.30	16.78	0.16	3.16	127.80	135.10	7.30	12.78	0.10	3.37
1270/17/W/–75/SE	186.80	231.00	44.20	40.42	0.20	3.69	186.80	231.00	44.20	41.58	0.20	4.03

1. Assay data missing from 219.30–303.70 m.
2. Drilled intersections – not true thickness.
3. Density by Archimedes method.

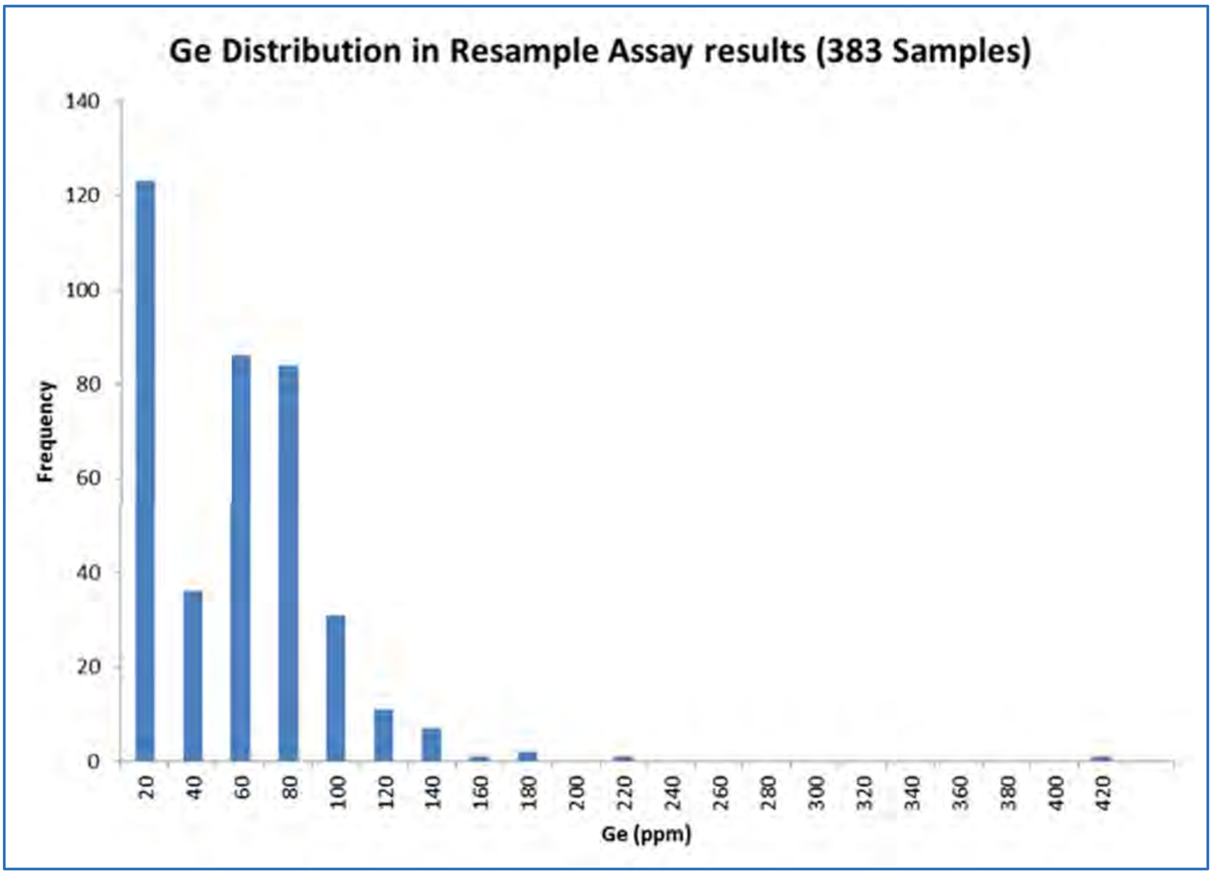
Figure 12.2 Scatterplot and Q-Q Plot Showing Gécamines Versus BVM Results for Zn





MSA, 2014

Figure 12.3 Histogram Plot of BVM Ge Results



MSA, 2014

12.2.4 Density Considerations

As part of the historical data verification exercise, density determinations were carried out by gas pycnometry on every tenth sample at BVM resulting in a data set of 40 readings. In addition, density determinations using the Archimedes method were carried out on a representative piece of 15 cm core for each sample during the 2013 re-logging campaign.

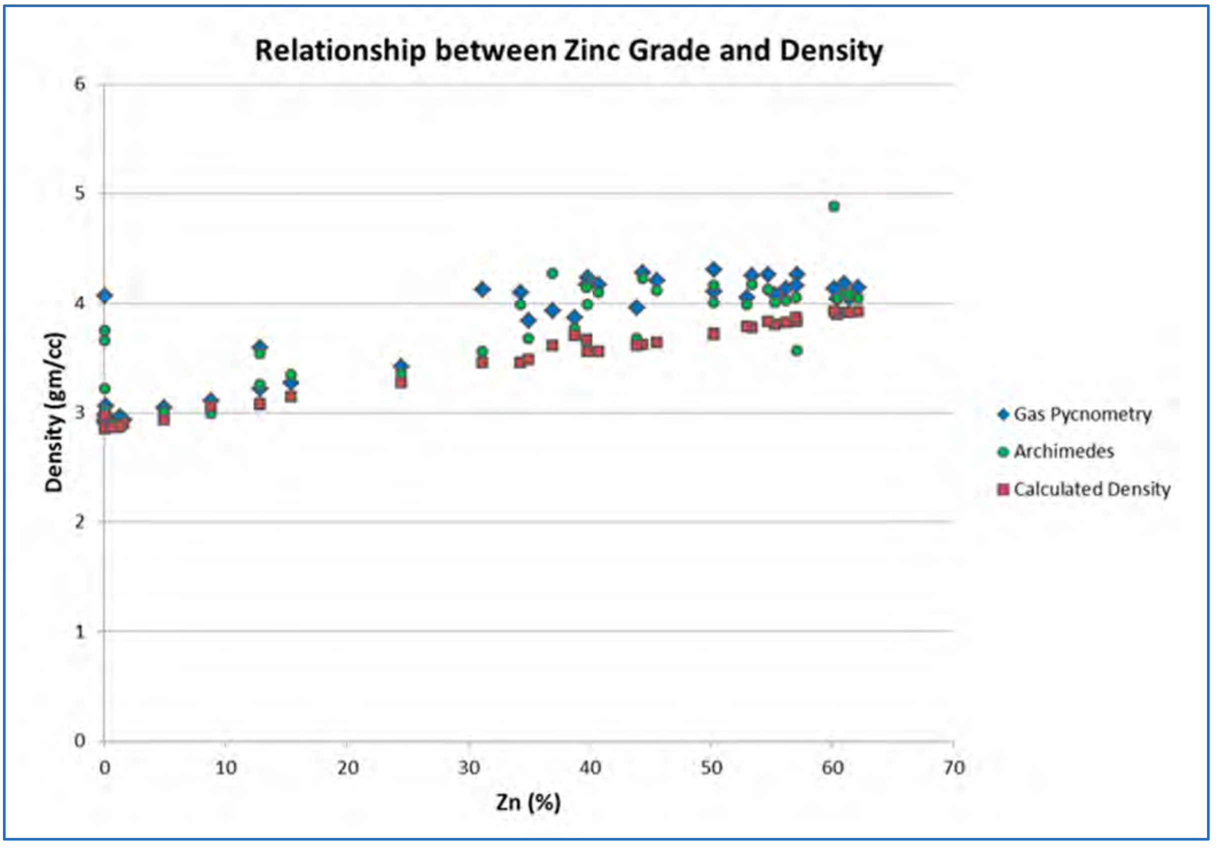
Gécamines used the following formula, derived mainly for the Fault Zone, to calculate density for use in its tonnage estimates:

$$\text{Density} = 2.85 + 0.039\ (\%Cu) + 0.0252\ (\%Pb) + 0.0171\ (\%Zn).$$

A comparison between density results based on the Gécamines formula, laboratory gas pycnometry method and the water immersion (Archimedes) method versus zinc grade for the same samples is shown in Figure 12.4. It is apparent that density, and hence tonnage, is understated by an average of 9% using the Gécamines calculated approach.

For the KICO drillholes, density was measured by KICO on whole lengths of half core samples using Archimedes principal of weight in air versus weight in water. Not all of the KICO samples were measured for density. A regression was formulated from the KICO measurements in order to estimate the density of each sample based on its grade. This formula was applied to the Gécamines samples and those KICO samples that did not have density measurements.

Figure 12.4 Relationship between Zn Grade and Density Calculated using the Gécamines Formula Versus BVM Laboratory Determinations by Gas Pycnometry and Archimedes Method Determinations



MSA, 2014

12.3 Re-logging Programme

KICO geologists undertook remarking and re-logging of all the available Gécamines drillholes that intersected the Big Zinc, using standardised logging codes which were also used in the KICO underground drilling programme.

12.4 Twin Hole Drilling Programme

Eleven Gécamines holes were twinned during the KICO underground drilling programme. The twin hole pairs are listed in Table 12.4, and examples of strip log comparisons between twin hole pairs are shown in Figure 12.5 to Figure 12.10.

In certain holes (e.g. 1270/7/V+30/–75/SE), Gécamines sampling stopped in mineralisation and complete sampling of the KICO twin holes allowed for determining the limits of mineralisation (Figure 12.9).

The KICO drillholes were more completely sampled in lower-grade mineralisation compared to the Gécamines holes as approximate visual cut-offs of 7% Zn and 1% Cu were used to guide the Gécamines sampling.

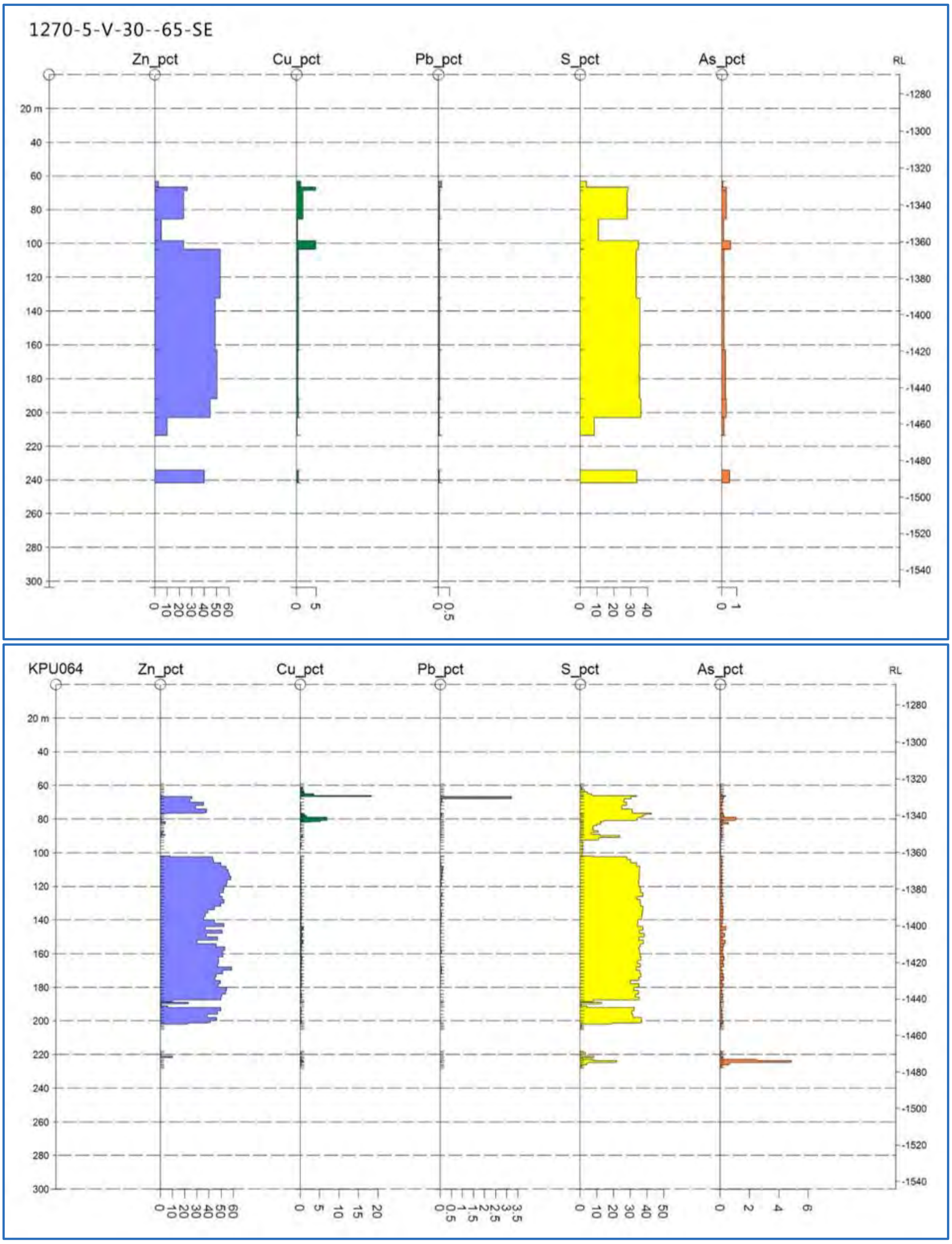
Sampling by KICO was initially carried out on a 1 m nominal length and later increased to 2 m, with sample length also constrained by lithology and mineralisation. More detail and grade resolution are observed in the KICO sampling compared to Gécamines sampling where sample lengths were based on homogenous zones of mineralisation ranging from less than 1 m to greater than 10 m in length with an average sample length of 3.44 m.

In general, the zinc, copper, and lead values compared well overall between the twin holes and the original holes.

Table 12.4 Kipushi Twinned Holes

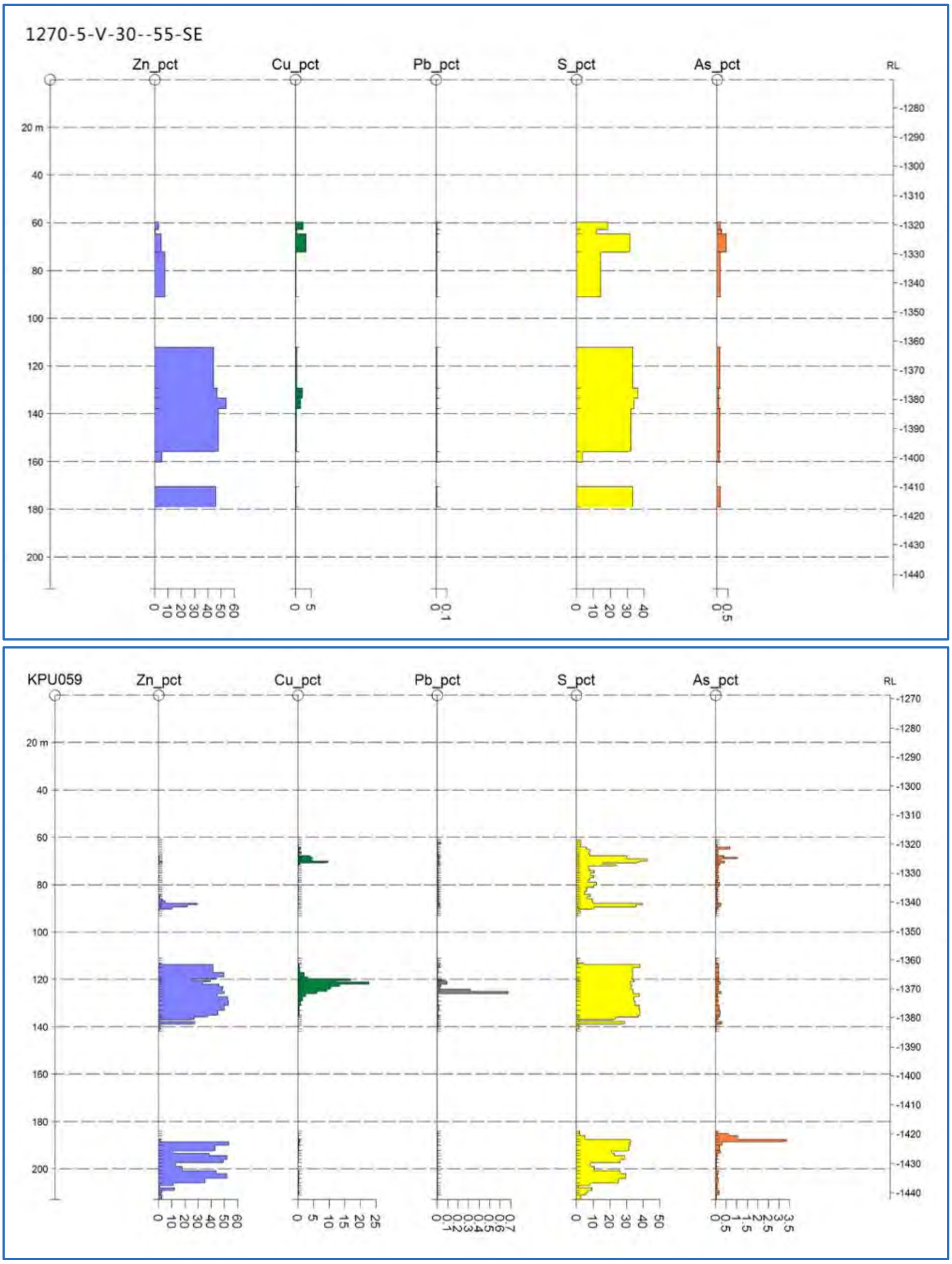
Gécamines Drillhole	Twinned with KICO Drillhole
1270/5/V+30/–45/SE	KPU046
1270/5/V+30/–65/SE	KPU064
1270/11/V+30/–65/SE	KPU062
1270/5/V+30/–55/SE	KPU059
1270/17/W/–35/SE	KPU070
1270/17/W/–76/SE	KPU069
1270/5/V+30/–75/SE	KPU057 and KPU051
1270/15/W/–20/SE	KPU068
1270/7/V+30/–75/SE	KPU051
1270/9/V+30/–63/SE	KPU071
1270/13/V+45/–30/SE	KPU065

Figure 12.5 Comparison between Gécamines Hole 1270/5/V+30/–65/SE and KICO Twin Hole KPU064



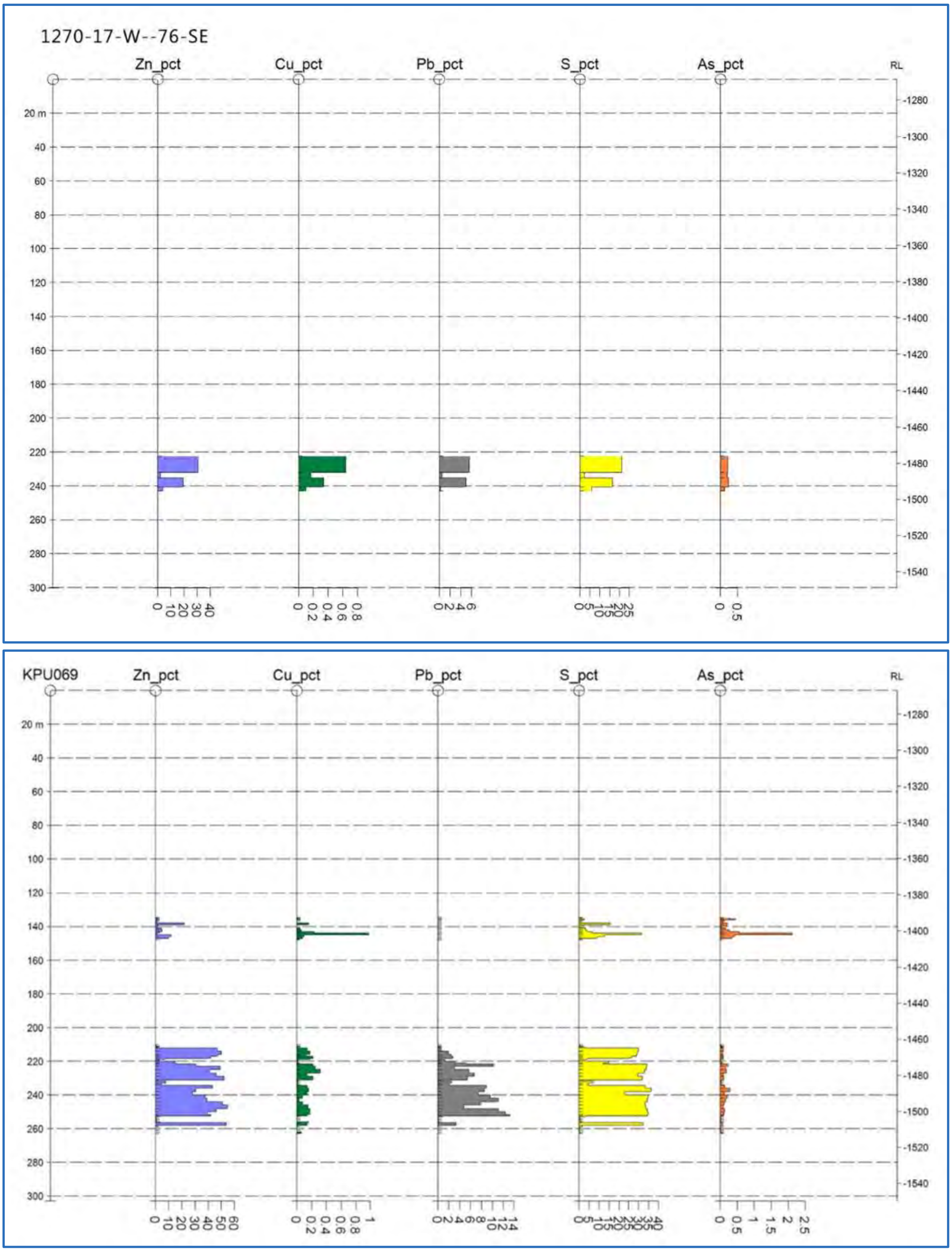
MSA, 2014

Figure 12.6 Comparison between Gécamines Hole 1270/5/V+30/–55/SE and KICO Twin Hole KPU059



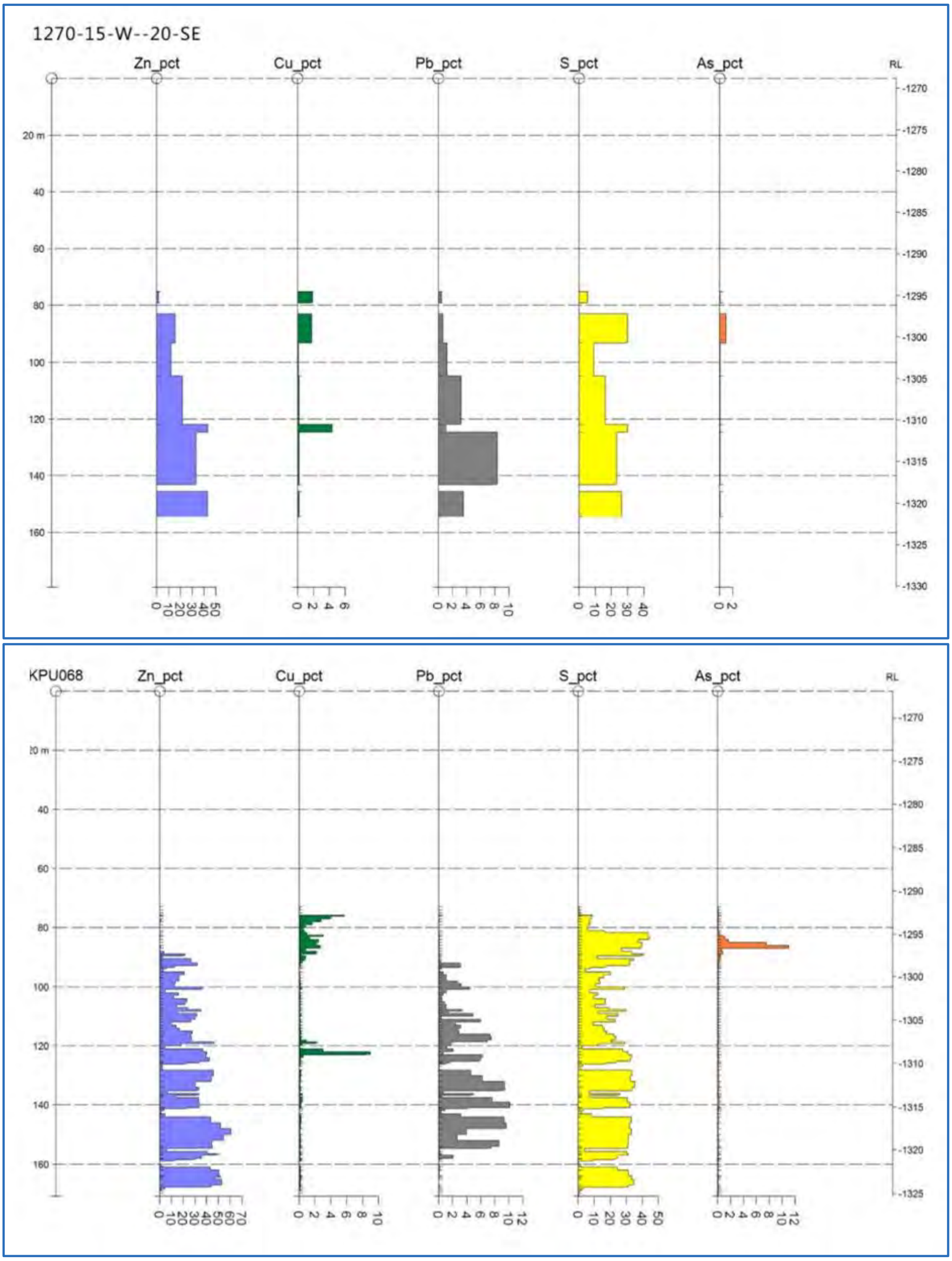
MSA, 2014

Figure 12.7 Comparison between Gécamines Hole 1270/17/W/–76/SE and KICO Twin Hole KPU069



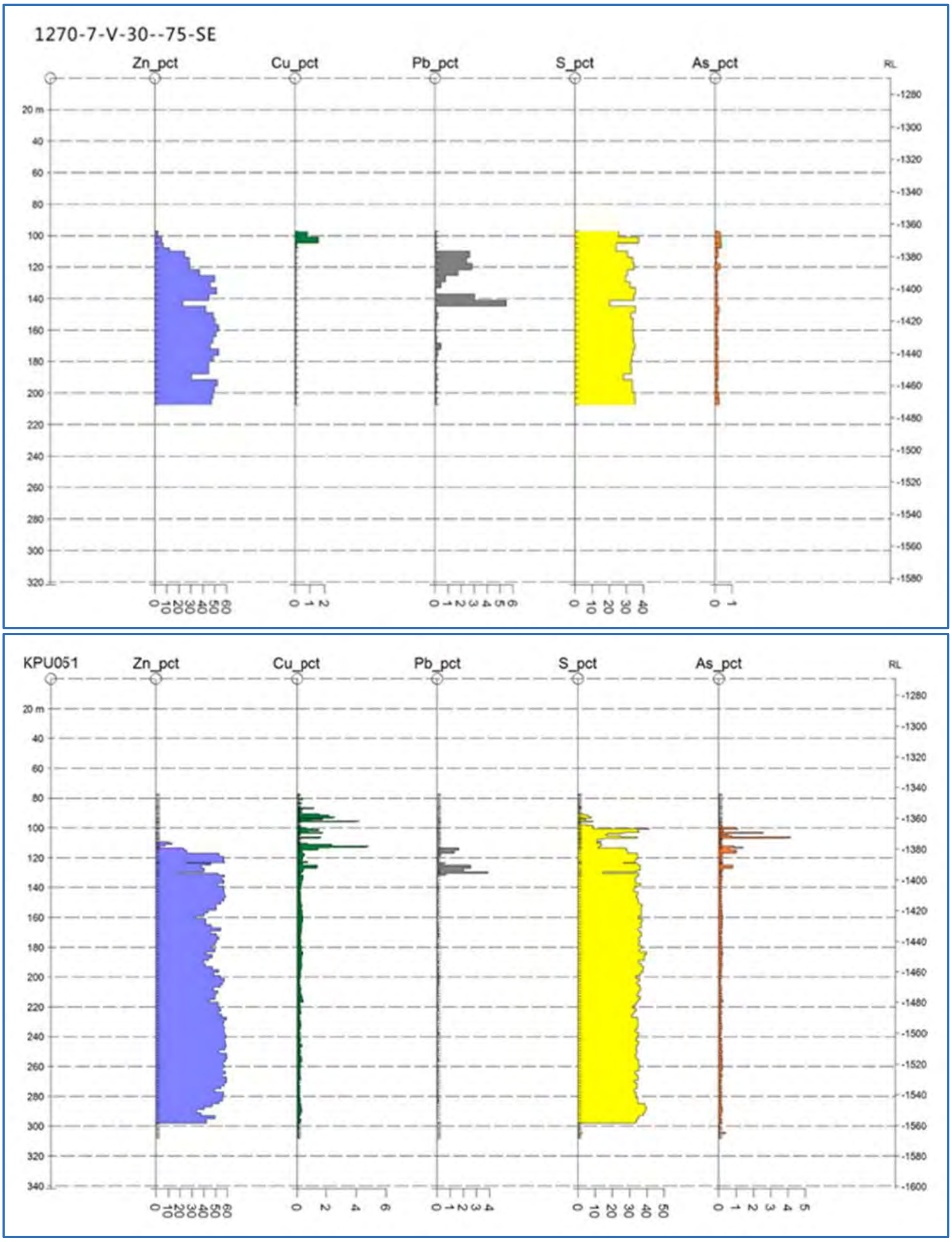
MSA, 2014

Figure 12.8 Comparison between Gécamines Hole 1270/15/W/–20/SE and KICO Twin Hole KPU068



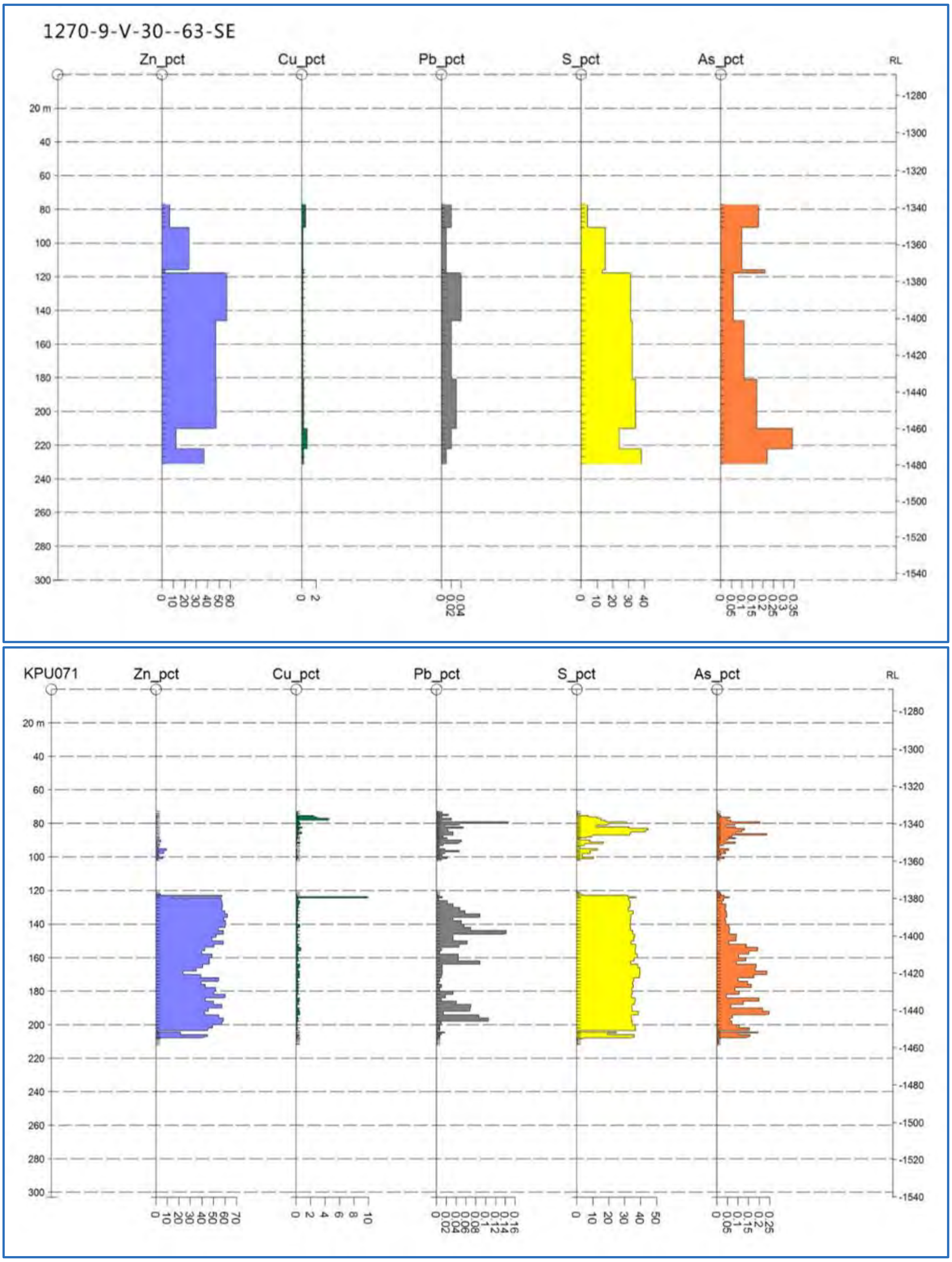
MSA, 2014

Figure 12.9 Comparison between Gécamines Hole 1270/7/V+30/–75/SE and KICO Twin Hole KPU051



MSA, 2014

Figure 12.10 Comparison between Gécamines Hole 1270/9/V+30/–63/SE and KICO Twin Hole KPU071



MSA, 2014

12.5 Visual Verification

Mineralisation in selected Gécamines and KICO drillholes was observed by the MSA QPs and compared against the assay results for these holes. It was concluded that the assays generally agree well with the observations made on the core.

12.6 Data Verification Conclusions

In the opinion of the MSA Geology QP, the results of the core re-sampling programme confirm that the assay values reported by Gécamines are reasonable and can be replicated within a reasonable level of error by international accredited laboratories under strict quality assurance and quality control (QA/QC) control.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

This section of the report defines the mineral processing and metallurgical testing for the Kipushi zinc project feasibility study incorporating a new zinc recovery process plant design. The Kipushi zinc project in the Democratic Republic of Congo (DRC) is being evaluated by Kipushi Corporation SA (KICO), with the objective of re-opening the mine and concentrator plant in the near future to produce saleable Zinc (Zn) concentrate.

The historic Kipushi processing plant comprised crushing, milling, flotation and concentration, and was in continuous operation from the late 1920s until the mine's closure in 1993. The main products from the mine were reported as zinc and copper concentrates. The mine also produced lead, cadmium, and germanium during this period.

Metallurgical testwork programmes were completed on drill core samples of known Kipushi mineralisation between 2013 and 2018 for the various project redevelopment study phases. These investigations were focused on metallurgical characterisation, and flowsheet development, for the processing of material from the Big Zinc orebody. The first test programme in 2013 included mineralogy, comminution and flotation testing. The second programme in 2015 was aimed to examine dense media separation (DMS). A review of potential process routes was then undertaken by KICO. The review suggested that given the favourable density differences between massive sulfides and gangue material, DMS was considered an alternate to flotation, potentially providing lower capital and operating costs, and this formed the basis for the Kipushi 2016 Preliminary Economic Assessment (PEA).

A third testwork programme was conducted between 2016 and 2018 on dedicated metallurgical drill core intercepts which were constituted into variability composites and a flow sheet development composite. Gravity separation (DMS and Spiral) efficiently rejected low SG dolomite. However, all sulfides in the feed report to the concentrate, reducing the concentrate zinc grade well below target requirements. A process flow sheet with milling and flotation circuits was thus considered for further upgrading of the DMS concentrate. The testwork results indicated that a saleable final flotation concentrate product at a Zn grade exceeding 53% could be achieved with an overall Zn recovery above 90%.

13.1 Previous Metallurgical Studies

13.1.1 Kipushi 2016 PEA

In 2016, a Preliminary Economic Assessment (PEA) study was executed which focussed primarily on the processing of the Big Zinc utilising gravity processes. It was proposed to process the ore in three size fractions through DMS and spiral facilities. A coarse DMS facility was to process 20 mm +6 mm with –6 mm +1 mm processed in a fine DMS. The –1 mm fraction was upgraded by spirals.

Two sets of metallurgical testing were conducted in support of the PEA study. These included:

13.1.1.1 Kipushi 2013 Scoping Study

A testwork campaign in 2013 was executed on a 60 kg sample derived from a composite of quarter core pieces. The average grade of the sample was 38% Zn, 12% Fe, 0.78% Pb and 0.4% Cu. The sample was predominantly sphalerite and pyrite, with minor amounts of galena and chalcopyrite. Scoping work that was conducted on this sample was promising and prompted the receipt of an additional fresh sample for testing. Highlights of this campaign were as follows:

- Preliminary comminution testing indicated that the composite was soft in nature, with Bond Ball Work Index (BBWI) of 7.8 kWh/t, and an Axb value of 105.
- Preliminary flotation testing at a grind of 80% passing 106 µm was promising and indicated that a rougher recovery of 87% Zn could be attained at a concentrate grade of 56% Zn.

13.1.1.2 Resource Development Phase – 2015

In 2015, a fresh metallurgical sample was obtained for further testing as part of resource model development. The sample comprised of six drill cores intercepting the Big Zinc and represented the major mineralisation types. The target head grades for the composite was 37% Zn. The major economic minerals within the composite were sphalerite (67%), galena (2%) and chalcopyrite (1%). Predominant non-sulfide gangue minerals included dolomite and quartz, with pyrite representing the major sulfide gangue phase. The scope of work incorporated:

- Gravity separation scoping work, which tested if the ore could be upgraded solely by gravity techniques. The circuit tested was the standard DMS/spirals flow sheet. The following parameters were investigated:
  - Effect of top size on heavy liquid separation (HLS) performance at three top sizes (–20 mm, –12 mm and –6 mm).
  - Effect of cut point on Zn recovery and final product grade, at a coarse size (HLS testing). This included establishment of washability curves to predict performance at various cut points.
  - Grade/recovery profiles (derived from shaking table work) demonstrating the potential to upgrade the derived fine fraction (–1 mm) via spirals.

The outcome of this study was that the composite could be satisfactorily be upgraded at a coarse size fraction, utilising DMS, where stage recovery in the region of 99% Zn was attained at concentrate grades of 55% Zn, at a cut point of 3.1 t/m³. It was noted that this result was consistently obtained at all three size fractions tested. The processing of the fines fraction was less successful, resulting in a stage recovery of 58% of Zn in feed at a final grade of approximately 56% Zn. As such, the metallurgical performance of the sample with the coarsest top size was marginally better than other top sizes, purely because the ratio of coarse to fine material in the feed was less than other samples tested.

Overall metallurgical performance for the final flow sheet was an overall Zn recovery of 95.4%, with a final concentrate grade of 55.5% Zn. This formed the basis for the PEA process flow sheet comprising:

- Two stage crushing with scrubbing and screening.
- DMS facility (–20 mm +6 mm, –6 mm +1 mm).
- Desliming of the –1 mm fraction, followed by spiral concentration.

13.1.2 Kipushi 2017 Pre-feasibility Study

Early into the PFS, it was identified that the proposed circuit had some limitations in terms of performance stemming from the gangue mineralogy. It was established that:

- The gravity circuits were highly effective at rejecting non-sulfide gangue, such as dolomite, as the density differential between the ore and non-sulfide gangue was large. As such, recovery to the final concentrate for the DMS was excellent.
- The gravity processes, however, served to concentrate the sulfides, and as such, pyrite, a predominant gangue phase in the ore would not be rejected in either the spirals or the DMS.

As a result, the concentrate grades obtained by both DMS and spirals would be driven by the proportion of pyrite in the feed, and through variability testwork and METSIM simulations conducted at the time. It was demonstrated that the required saleable concentrate grade would not be attainable by this method.

The process flow sheet evolved from the proposed gravity circuit to use of hybrid DMS/flotation circuits in order to provide a suitable mechanism for pyrite rejection.

The following flow sheet options were proposed:

- The PEA flow sheet – referred to the Gravity Flow sheet, as described above.
- The DMS/Float process – which entails preconcentration of the coarse ore fraction by DMS, followed by sequential rougher flotation of the combined –1 mm fraction and DMS concentrate to produce a saleable Zn concentrate. The proposed sequential flotation circuit comprised of:
  - Stage 1: Depression of sphalerite and pyrite by NaCN; $ZnSO_4$ solution at elevated pH to recover Cu/Pb concentrate.
  - Stage 2: Re-activation of sphalerite using $CuSO_4$ solution, combined with raised pH to depress remaining pyrite.
- The run-of-mine (ROM) Float process – which entails the recovery of high-grade Zn concentrate via above sequential flotation regime, using a standard MF1 (mill/float) configuration applied to ROM ore.
- A very brief investigation into the use of a bulk sulfide flotation regime was undertaken, and, as this showed promise, was investigated further in the Feasibility Study (FS) phase of the project.

Testwork to support economic evaluation of these options was conducted on a new master composite. This composite was made up of four drill cores intercepting the Big Zinc, selected to represent all styles of mineralisation present in the orebody. The assayed intervals were composited in such a manner that a master composite with a head grade of 32–33% Zn could be produced. This grade range represented the average ROM Zn grade, based on the Pre-feasibility Study (PFS) mine plan.

The scope of work included:

- Ore characterisation work on the master composite (bulk mineralogy, mineral mode of occurrence, association and liberation characteristics, Electron Microprobe Analysis (EMPA), chemical analysis). It should be noted that the mineralogy was conducted on a –1 mm feed sample.
- Comminution testwork on four composited (Bond Ball Work Index (BBWI), Bond Rod Work Index (BRWI)).
- Gravity separation testwork, including:
  - HLS testwork conducted at a –20 mm +1 mm size fraction. The test was conducted at a SG range from 2.7–3.8 t/m^3 in 0.1 increments.
  - Shaking table testwork on the fines fraction deslimed at 38 µm ahead of testing. The standard shaking table test arrangement was utilised, where nine distinct gravity fractions were collected in order to present grade/recovery profiles across the table.
- Flotation testwork was conducted on both ROM samples and samples comprising the DMS concentrate combined with the fines fraction. The testwork campaign included optimisation of grind size, as well as some reagent, and process condition optimisation. Most work was executed as rougher rates tests. However, some alternative flotation strategies were also investigated, including:
  - Sequential flotation: Use of cleaner stage on Cu/Pb rougher concentrate to improve overall Zn recovery.
  - Bulk sulfide flotation on rougher scale was executed to compare with the sequential flotation route, as this would have a simpler reagent suite, and could result in operating costs savings if metallurgically similar performance could be obtained.

13.1.2.1 Mineralogical Study

The major mineral species present, in order of prevalence, were sphalerite, dolomite, pyrite, quartz, galena, and chalcopyrite. Bulk mineralogy conducted indicated that the PFS master composite has a significantly lower sphalerite content when compared to previous samples tested (49.37% vs 66.8%), with an increased pyrite content (13.56% compared to 7.8%). The increased pyrite content contributed significantly to dilution of Zn grade of gravity concentrates.

EMPA analysis indicated that the sphalerite grains had experienced some Fe substitution and that grains (n=84) assayed 1.47% Fe on average. This suggests a typical sphalerite composition of Zn 0.975 and Fe 0.025S. This level of substitution is relatively low compared to many globally important zinc ores with compositions ranging from 3.2% Fe to 8.1% Fe (Lockington, JA, 2012, grains probed =362, orebodies =6, core interval average values giving quoted range n=19). In the case of Broken Hill, the high iron (and manganese) content leads to a black sphalerite colour and the term Marmatite is used as a substitute for Sphalerite. As the Kipushi zinc sulfide is very low in iron the term sphalerite is appropriate.

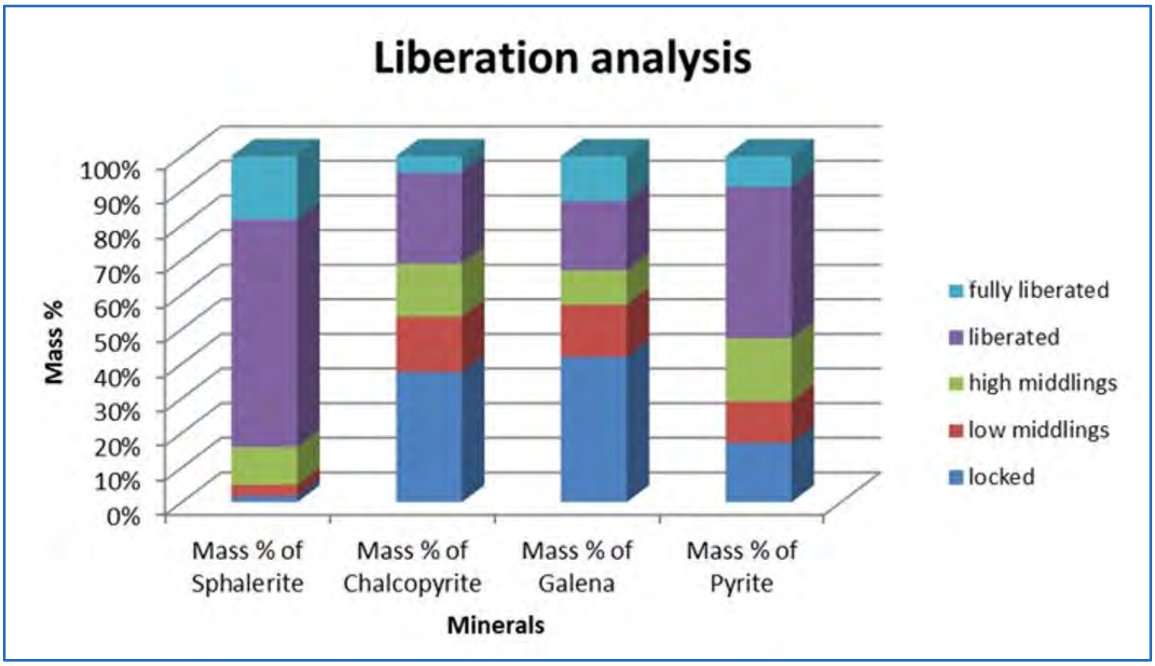
It was also noted that none of the mineral types identified posed an occupational health and safety risk.

The sphalerite present in the PFS master composite was relatively coarse grained. Galena and chalcopyrite are both medium to fine grained, with approximately 50–60% of these mineral phases being finer than 40 µm.

Mineral association and liberation were examined at a very coarse crush size (100% passing 1.7 mm) to assist the evaluation of gravity separation. Chalcopyrite and galena have similar liberation characteristics and, when not liberated both were associated with sphalerite. Pyrite had better liberation characteristics than chalcopyrite or galena and, when unliberated, was dominantly associated with sphalerite.

Sphalerite was noted to have a high degree of coarse liberation, with approximately 85% of grains fully or highly liberated in this 1.7 mm top size sample (Figure 13.1). However, 40% of galena, and 38% of chalcopyrite were classified as 'locked', contained in grains where they made up 20% or less of the mass.

Figure 13.1 AUTOSEM Liberation Analysis of –1.7 mm Fractions



METC, 2022

13.1.2.2 Comminution Testwork

BRWI testing was conducted at a limiting screen of 1.18 mm and indicated that the four composites tested were characterised as soft, with indices ranging from 7.4–13.4 kWh/t, with an 80th percentile value of 10.9 kWh/t.

BBWI testing was conducted at a limiting screen of 106 µm and indicated that the four composites tested were characterised as soft, with indices ranging from 7.72–9.12 kWh/t, with an 80th percentile value of 9.2 kWh/t.

13.1.2.3 Gravity Separation Testwork

HLS testwork on the PFS master composite confirmed the sample received was amenable to upgrade via DMS and showed potential for rejection of a barren gangue discard. At the proposed cut point of 3.1 t/m^3, it was recorded that mass rejection of approximately 30% was attainable, with a very low impact on Zn recovery (1%). The mass rejected was predominantly dolomite, with upgrading noted for all sulfide species present. The sample was upgraded from a head grade of 34.1%–49.90% Zn at this cut point. The testwork also indicated that at higher cut points, very little additional increase in Zn concentrate grade was attainable, with substantial losses in Zn recovery.

It was noted that DMS concentrate grades more than 50% Zn could not be attained with the sample tested.

As was previously observed, shaking table testwork on the naturally occurring fines derived from the PFS master composite could be upgraded to 50% Zn, but at recoveries below 77% Zn.

The results confirmed that gravity separation alone was unable to consistently generate suitable saleable concentrates.

13.1.2.4 Flotation Testwork

The traditional means of processing Cu-Pb-Zn ores is to perform a sequential float, where the Zn within the ore is initially depressed to produce a Cu/Pb rougher concentrate. After this, the Zn is re-activated and is recovered in a second rougher flotation stage. In the Cu-Pb rougher flotation circuit, the pH is typically elevated to 9.5, utilising Soda Ash as a pH modifier. This is because lime has been known to activate sphalerite flotation. NaCN is utilised in combination with $ZnSO_4$ (3–5 times the amount of cyanide) to depress the sphalerite. Collection of the galena and chalcopyrite is conducted utilising a highly selective xanthate, or alternative collector. Once the Cu-Pb rougher stage is completed, the Zn is re-activated utilising $CuSO_4$. Lime is utilised as a pH modifier in this stage.

As the Zn feed grade of the ore processed for the Kipushi Project is high, a less complex process route, where all sulfides are recovered as a bulk concentrate, was evaluated as an alternative. Bulk sulfides are floated using xanthate collector in conjunction with $CuSO_4$ as an activator.

Both options were tested on two feed scenarios, namely:

- A flotation feed comprising ROM ore, and
- A flotation feed comprising DMS concentrate combined with the fines fraction, referred to as a pre-concentrated ore sample.

In addition to evaluating the metallurgical performance of the above-mentioned flow sheets, the programme established optimal grind, residence time, and executed a small reagent optimisation programme.

The outcomes of these tests were as follows:

- Testwork was conducted at two grinds, to establish optimum grind, P_{80} of 106 µm and 150 µm respectively. It was observed during this testing that the coarser grind displayed faster kinetics in the initial phases of testing, but that overall Zn recovery at the end of the test was lower compared to the finer mill product. Based on this observation, the optimal grind was determined to be 80% passing 106 µm.
- The appropriate laboratory residence times were established as follows:
  - Sequential flotation – Cu-Pb flotation ~4 minutes, Zn flotation ~12 minutes, and
  - Bulk flotation ~10 minutes.
- Despite optimisation, the Zn losses to the Cu-Pb reject (first stage froth product) were substantial when applying sequential flotation to both pre-concentrated feed and ROM ore respectively. This is thought to be due to the high degree of association of sphalerite with galena and chalcopyrite.
  - It was observed that recoveries of 92% Zn at a concentrate grade of 58% could be attained for the pre-concentrated feed, compared to 95% Zn recovery at a concentrate grade of 54% Zn for the ROM feed.
  - These were based on standard rougher kinetic flotation testing. It was noted that Zn recovery losses to the Cu-Pb concentrate could be reduced by 3% through reverse cleaning of the Cu-Pb concentrate.
  - Bulk sulfide flotation of the preconcentrated feed (single PFS test only) produced an overall recovery of 97–99% Zn at a final concentrate grade of 56% Zn.
- Bulk sulfide flotation of the preconcentrated feed produced an overall recovery of 97.4–99% Zn at a final concentrate grade of 56% Zn. A single test was conducted, and no further evaluation was completed during the PFS phase of the project.

13.1.2.5 Kipushi 2017 PFS Flow Sheet Selection

The Kipushi 2017 PFS programme demonstrated that the proposed coarse/fine gravity separation-based circuit would not be able to produce concentrates above the 53% Zn grade threshold. This is because gravity separation processes were limited to removing non-sulfide gangue from the ore. This approach is only applicable to samples with high Zinc head grades and minor proportions of sulfide gangue.

The use of flotation was, therefore, essential to attain the necessary upgrades. The introduction of the bulk sulfide flotation regime was very late in the PFS, and as such, the PFS was based on a sequential flotation process flow sheet. However, as bulk sulfide flotation was also successful it was recommended that this be evaluated further in the FS.

A trade off study was conducted to establish if any financial benefit could be obtained by pre-concentrating the ore by DMS ahead of the flotation circuit. The inputs to this study were the metallurgical test outputs, as well as CAPEX and OPEX estimates for each option. The techno-economic study was conducted by KICO and demonstrated that the most economically viable solution would be to preconcentrate the ore ahead of flotation by DMS.

13.1.2.6 Overall Metallurgical Performance (as tested)

Based on the testwork results and the outcomes of the techno-economic study, the metallurgical inputs into the PFS are presented in Table 13.1.

Table 13.1 Kipushi PFS Metallurgical Inputs

Description	Units	Value
DMS Plant		
DMS Cut Point	t/m^2	3.1
DMS mass pull (stage)	%	30
Overall Zn losses to DMS floats (reject)	%	0.5
Flotation Plant		
Grind (P_{80})	µm	106
Reagent Regime (Cu-Pb Circuit)		
Operating pH		9.5
pH Modifier		Soda Ash
pH Modifier Dose	g/t	800
Depressant (NaCN) Dose	g/t	400
Depressant ($ZnSO_4$) Dose	g/t	800
Collector (SEX) Dose	g/t	20
Frother (MIBC) Dose	g/t	12
Reagent Regime (Zn Circuit)		
Operating pH		11.5
pH Modifier		Lime
pH Modifier Dose	g/t	1,600
Collector (SIPX) Dose	g/t	40
Activator ($CuSO_4$) Dose	g/t	1,800
Frother (MIBC) Dose	g/t	38
Flotation Performance		
Overall Zn Recovery	%	90.2
Final Zn Concentrate Grade	%	58.9

13.2 Feasibility Study Metallurgical Testwork

13.2.1 Testwork Goals and Objectives

The testwork programme had the following objectives / required outcomes:

- Expansion of the existing comminution data set to include SAG milling comminution testing (SMC) testing, Uniaxial compressive strength (UCS) testing and Bond Crusher Work Index (Cwi) testing. This programme included a variability study comprising nine samples of the three main ore types in the deposit.
- Confirmatory flow sheet testing on a master composite, representative of the average grade, and mineralogical composition of the ore over the project life-of-mine (LOM). Scope of work included HLS characterisation and baseline flotation testing on ROM and preconcentrated ore. This series of testing included use of a sequential flotation regime, as well as a bulk sulfide flotation regime.
- Flotation process development and optimisation studies, including flotation of ROM and preconcentrated ore. Testwork also focussed on optimising conditions for bulk sulfide flotation, with focus on pyrite rejection.
- Investigation of the effect of site water quality on flotation performance.
- Bulk concentrate production for concentrate characterisation, and to generate sample for vendor testing.
- Vendor testwork (thickening and filtration) for equipment sizing requirements.

13.2.2 Sample Selection and Provenance

13.2.2.1 Comminution Testwork Samples

Nine comminution samples were prepared from drill core samples for comminution testwork. The samples were selected to represent the major ore types identified within the Big Zinc deposit. The final sample set consisted of three samples from Dolomite (SDO) ore, three samples of Mixed Sulfide Minerals (MSM) ore, two samples of MBS, and a single sample of MRS ore. The samples were derived from three metallurgical drill cores, KPU100, KPU101, and KPU104. A composition of each composite is presented in Table 13.2.

Table 13.2 Sample Provenance – Comminution Variability Samples

Sample Description	Core	Sections
KPU101SDO	KPU101	103.3 m–108.71 m
KPU104SDO1	KPU104	65.28 m–71.40 m
KPU104SDO2	KPU104	334.85 m–340.10 m
KPU104MBS1	KPU104	205.12 m–210.12 m
KPU104MBS2	KPU104	315.10 m–320.30 m
KPU104MRS	KPU104	79.45 m–85.63 m
KPU101MSM	KPU101	111.18 m–123.28 m
KPU104MSM	KPU104	178.74 m–184.43 m
KPU100MSM	KPU100	127.38 m–132.56 m

The location of these sections relative to the orebody are presented in Figure 13.2.

13.2.3 DMS and Flotation Sample Selection and Provenance

13.2.3.1 Sample Provenance

Eighteen drill core intersections were selected for metallurgical testing based on their ore type. The drill core sections were derived from two metallurgical drill cores, namely KPU101 and KPU104. The 18 samples were composited based on their lithological ore type into 10 composites for use in testing, and the provenance of each sample received is presented in Table 13.3.

These composites were utilised to prepare the following samples:

- Feasibility development composite, which was utilised for flow sheet development and optimisation studies.
- Variability test samples.

Figure 13.2 Comminution Sample Locations with Respect to Big Zinc Deposit

Ivanhoe, 2018

Table 13.3 Sample Provenance – DMS and Flotation Core Sections

Composite	Core	Mass (kg)	Section length	Ore Type/ Lithology	Category
Composite 1	KPU101	22.99	152.36 m–158.27 m	SDO	Dolomite
	KPU104	31.06	49.43 m–65.28 m	SDO	Dolomite
	KPU104	24.52	90.50 m–105.85 m	SDO	Dolomite
	KPU104	34.33	143.04 m–151.65 m	SDO	Dolomite
Composite 2	KPU104	55.51	164.11 m–174.28 m	MBS	Ave Cu, High Zn, Ave Fe
	KPU104	40.44	184.64 m–192.10 m	MBS	Ave Cu, High Zn, High Fe
Composite 3	KPU104	45.62	155.10 m–164.11 m	MBS	Ave Cu, High Zn, High Fe
	KPU104	47.40	260.10 m–267.10 m	MBS	High Cu, High Zn, High Fe
	KPU104	48.00	278.00 m–285.10 m	MBS	Low Cu, High Zn, High Fe
Composite 4	KPU104	65.33	129.21 m–141.74 m	MBS	Low Cu, Ave Zn, Ave Fe
	KPU104	27.08	345.14 m–350.65 m	MBS	Low Cu, High Zn, Ave Fe
Composite 5	KPU101	30.48	108.71 m–118.18 m	MSM	Low Cu, Low Zn, Low Fe, High SDO
Composite 6	KPU101	3.54	84.65 m–92.18 m	MSM	N/A
Composite 7	KPU101	37.54	141.56 m–150.6 m	MSM	High Cu, Ave Zn, Low Fe
Composite 8	KPU101	26.50	123.96 m–129.07 m	MRS	Low Cu, High Zn, Ave Fe
	KPU104	38.74	39.90 m–47.05 m	MSM	Low Cu, High Zn, High Pb, Low Fe
Composite 9	KPU104	21.95	52.90 m–58.18 m	MRS	Low Cu, High Zn, Low Fe
	KPU104	23.20	342.85 m–353 m	MRS	Low Cu, High Zn, Low Fe
Composite 10	KPU104	24.97	174.28 m–178.74 m	MSM	High Cu, High Zn, High Fe

The chemical analysis of the 10 composites are presented in Table 13.4.

In addition to the 10 composites prepared for testing, an additional dolomite waste sample was prepared.

13.2.3.2 Feasibility Study Development Sample

The FS development composite was created from the 10 ore type composites generated for the Project. The proportions of each of the 10 ore type composites were based on obtaining a development composite representing the average grades of the key elements (Zn, Fe, Cu, Pb, and S) as per the Kipushi 2022 FS mine plan.

The composition of the FS development sample is presented in Table 13.5.

Table 13.4 Chemical Composition – FS Ore-Type Composite Samples

Sample Name	Grade (%)														
	Zn	Total S	Al	Ca	Co	Cr	Cu	Fe	Mg	Mn	Ni	Pb	Si	Ti	V
Composite 1	5.23	5.44	0.32	16.7	0.05	0.05	0.05	2.18	11.0	0.16	0.05	0.14	2.15	0.05	0.05
Composite 2	51.0	32.2	0.07	0.84	0.05	0.05	0.69	8.95	0.37	0.05	0.05	0.05	0.39	0.05	0.05
Composite 3	43.0	39.1	0.07	0.73	0.05	0.05	0.37	11.5	0.38	0.08	0.05	0.05	0.36	0.05	0.05
Composite 4	38.3	31.0	0.07	2.31	0.05	0.05	0.27	11.2	1.41	0.05	0.05	0.20	2.34	0.05	0.05
Composite 5	12.5	12.8	1.66	8.17	0.05	0.05	0.36	5.48	5.50	0.08	0.05	1.60	9.76	0.13	0.05
Composite 7	30.5	21.7	1.30	4.31	0.05	0.05	0.69	5.10	2.98	0.05	0.05	2.48	6.25	0.12	0.05
Composite 8	44.7	30.3	0.73	0.40	0.05	0.05	0.22	7.02	0.29	0.05	0.05	7.60	3.35	0.06	0.05
Composite 9	47.4	29.8	0.05	1.84	0.05	0.05	0.13	5.40	1.11	0.05	0.05	0.13	4.47	0.05	0.05
Composite 10	46.9	38.1	0.05	0.58	0.05	0.05	2.82	12.6	0.16	0.05	0.05	0.05	0.26	0.05	0.05
Dolomite composite	0.97	1.66	0.16	18.6	0.05	0.05	0.05	1.60	12.2	0.14	0.05	0.05	1.64	0.05	0.05

Table 13.5 Composting Strategy and Chemical Analysis for FS Development Composite

Sample Name	Mass (kg)	Mass (%)	Grade (%)								
			Zn	Total S	Ca	Cu	Fe	Mg	Mn	Pb	Si
Composite 1	80	16.6	5.23	5.44	16.7	0.05	2.18	11.00	0.16	0.14	2.15
Composite 2	70	14.5	51.0	32.2	0.84	0.69	8.95	0.37	0.05	0.05	0.39
Composite 3	105	21.7	43.0	39.1	0.73	0.37	11.50	0.38	0.08	0.05	0.36
Composite 4	65	13.5	38.3	31.0	2.31	0.27	11.20	1.41	0.05	0.20	2.34
Composite 5	12	2.48	12.5	12.8	8.17	0.36	5.48	5.50	0.08	1.60	9.76
Composite 7	24	4.97	30.5	21.7	4.31	0.69	5.10	2.98	0.05	2.48	6.25
Composite 8	40	8.28	44.7	30.3	0.40	0.22	7.02	0.29	0.05	7.60	3.35
Composite 9	27	5.59	47.4	29.8	1.84	0.13	5.40	1.11	0.05	0.13	4.47
Composite 10	10	2.07	46.9	38.1	0.58	2.82	12.60	0.16	0.05	0.05	0.26
Dolomite Composite	50	10.4	0.97	1.66	18.6	0.05	1.60	12.20	0.14	0.05	1.64
FS Composite Grade (Estimated)	483	100	32.0	24.8	5.84	0.36	7.36	3.78	0.08	0.87	2.06
FS Composite Grade (Measured)	–	–	29.6	24.5	5.88	0.32	8.49	4.08	0.07	0.82	2.17

13.2.3.3 Variability Sample Selection

The possibility of treating single ore types alone is remote, therefore, realistic plant feed variability samples, consisting of mixtures of the ore types in Table 13.4, were prepared. In addition, there is only limited opportunity for grade control blending through the ore handling train, so the expected range of feed compositions is broad. The basis for selection of variability testwork samples was the range of assays of Cu, Pb, Fe, and Zn expected over the Project LOM plan. The nine variability samples were prepared from the original 10 ore type composites.

The key variabilities influencing metallurgical performance are the proportions of pyrite, chalcopyrite, and galena in plant feed. Therefore, the selection of variability samples was influenced by the spread of grades for these elements in relation to the Zn grade over the LOM. Samples were selected to represent various combinations of Fe, Pb, Cu, and Zn within the monthly average LOM schedule (Refer to Figure 13.3, Figure 13.4, and Figure 13.5).

Figure 13.3 Grade Spread (Zn vs Fe) over LOM, including Variability Sample Selection

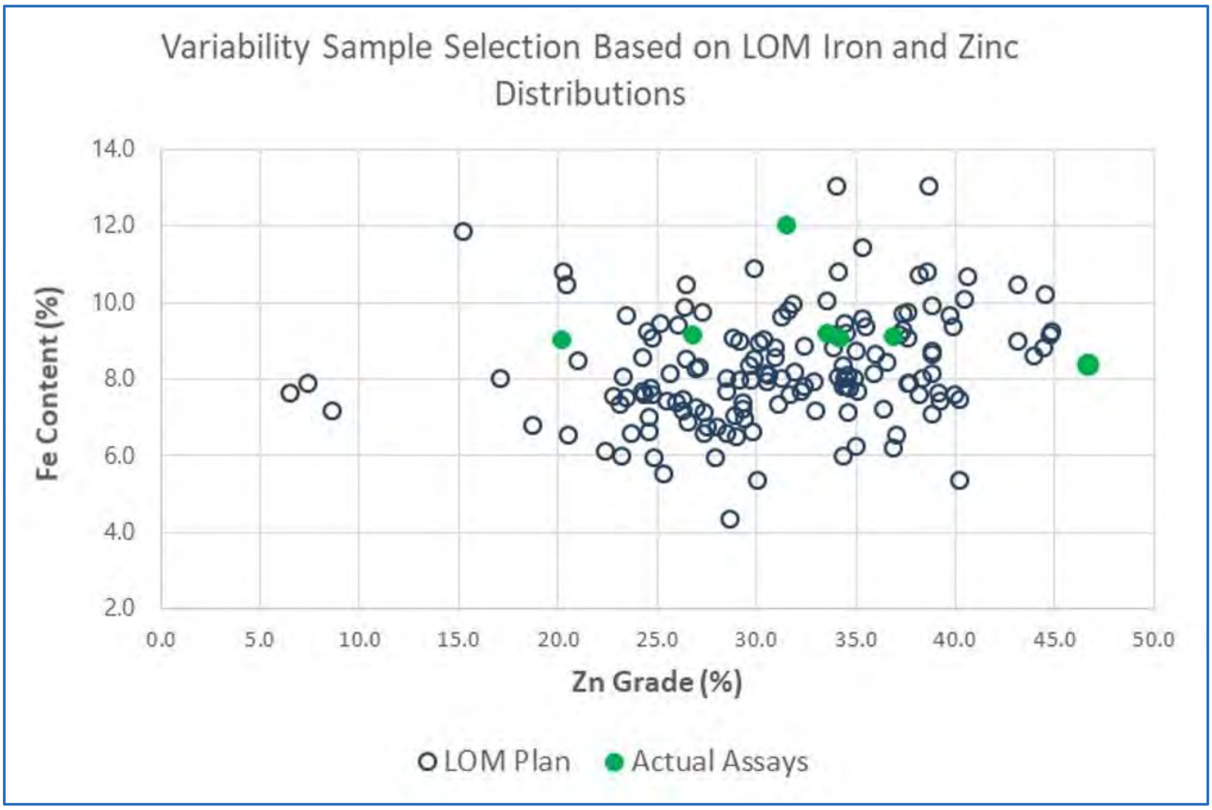


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Figure 13.4 Grade Spread (Zn vs Cu) over LOM, including Variability Sample Selection

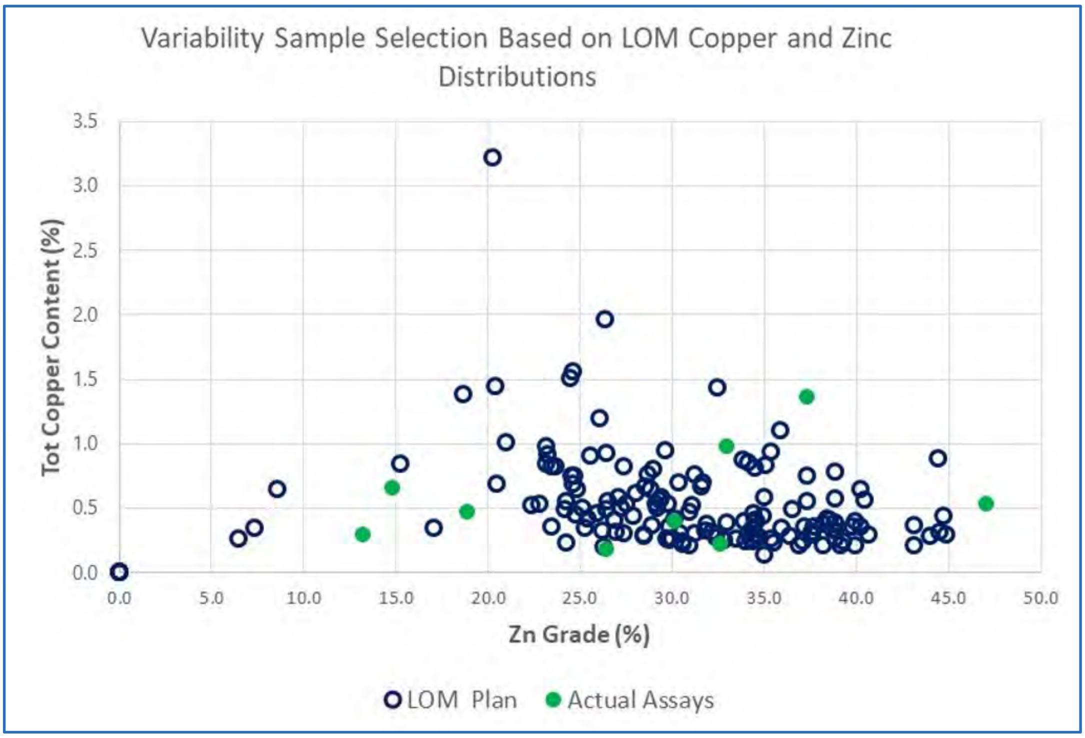

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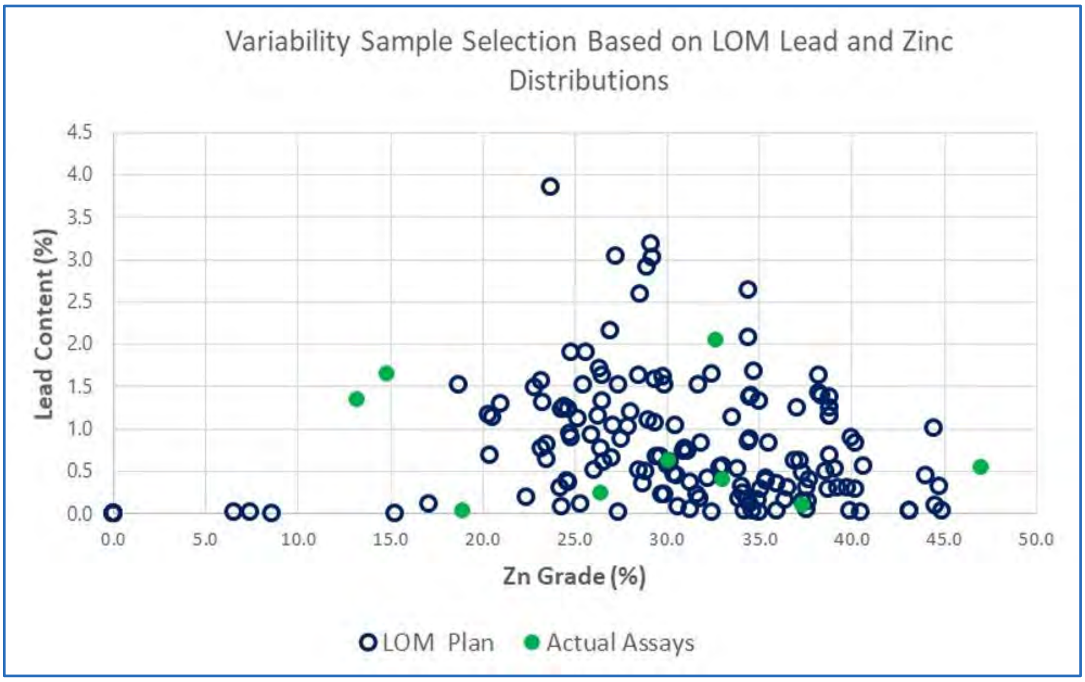
Figure 13.5 Grade Spread (Zn vs Pb) over LOM, including Variability Sample Selection



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It must be noted that the Fe grades (and therefore the pyrite contents) in the variability samples (Figure 13.5) are all at the average Fe grade or higher. Therefore, the variability flotation results can be expected to represent ore that is more difficult than average to upgrade.

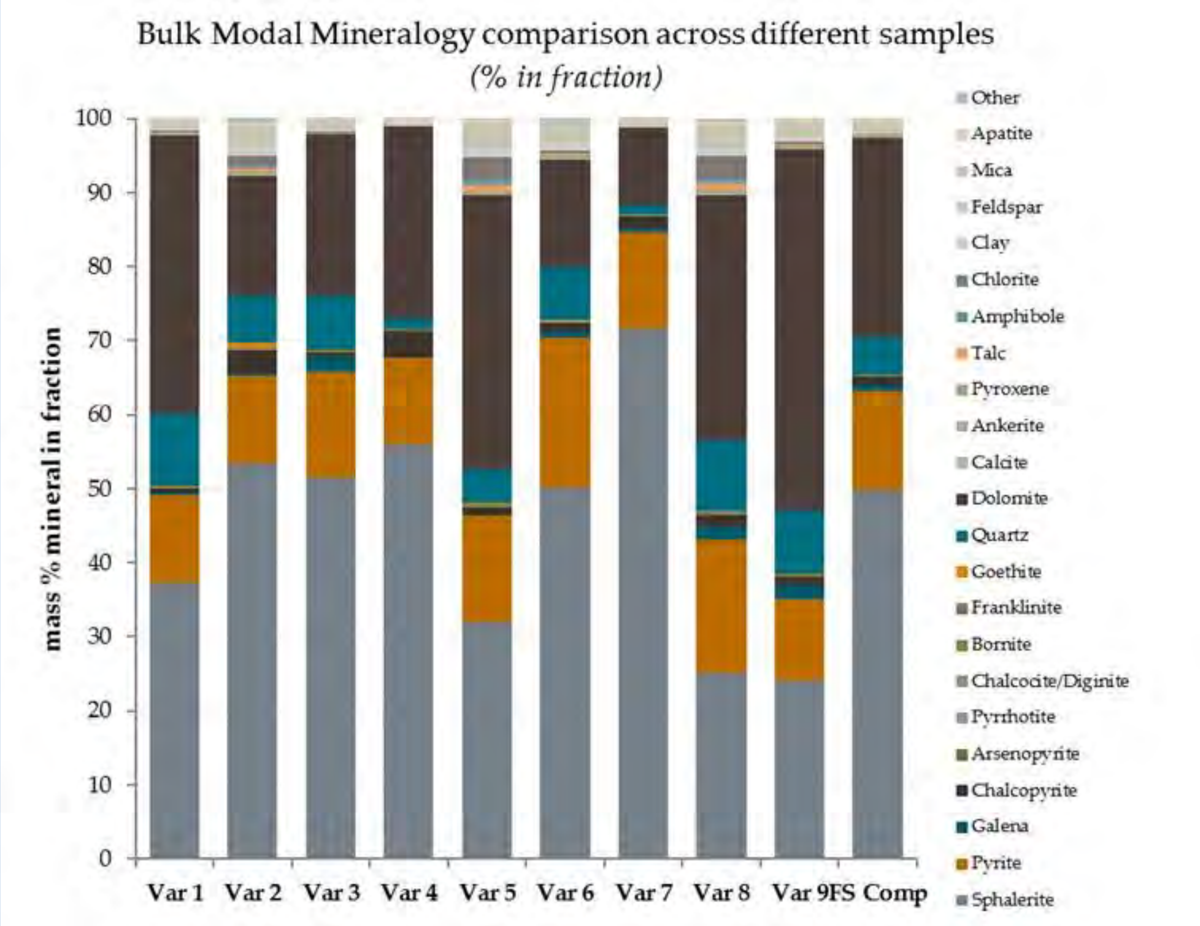
The head grade chemical analysis for each of the variability samples is presented in Table 13.6.

Table 13.6 Chemical Analysis of Variability Composites

Sample Name	Head grade (%)					
	Cu	Pb	Zn	Total S	$Fe_{(tot)}$	SiO_2
Variability 1	0.22	2.05	32.6	26.7	8.27	7.50
Variability 2	1.36	0.10	37.3	29.7	9.88	1.71
Variability 3	0.18	0.25	26.4	22.7	8.67	8.32
Variability 4	0.98	0.40	33.0	34.6	8.7	8.31
Variability 5	0.47	0.03	18.9	17.7	8.41	10.5
Variability 6	0.40	0.62	30.1	27.1	11.3	10.4
Variability 7	0.53	0.55	47.0	35.7	8.42	1.72
Variability 8	0.66	1.65	14.8	21.3	12.2	13.5
Variability 9	0.29	1.35	13.2	12.9	6.29	8.41

The bulk mineralogical composition (QEMSCAN analysis) of each of the variability composites is presented in Figure 13.6.

Figure 13.6 Bulk Mineralogical Composition of Variability Samples



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The following ranges were covered in terms of composition:

- Sphalerite ~24–72%,
- Pyrite ~11–20%,
- Dolomite ~10–48%,
- Galena ~0.02–2%, and
- Chalcopyrite ~0.6–4%.

13.2.4 Comminution Testwork

The scope of work for the comminution study was to establish the range of variability for the following comminution parameters:

- Uniaxial compressive strength (UCS),
- SAG milling comminution (SMC) testing,
- Bond Abrasion Index (Ai),
- Ball Rod Work Index (BRWi),
- Bond Ball Work Index (BBWi), and
- Bond Crushability Work Index (Cwi).

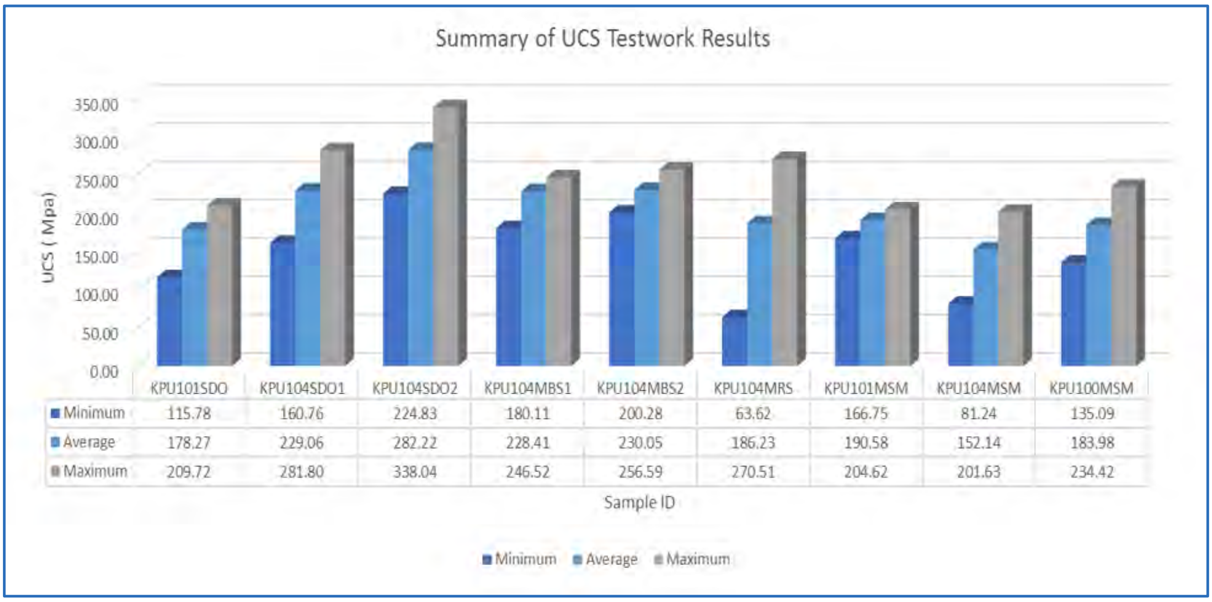
As described previously, nine comminution samples, comprising multiple samples of the various ore types present in the orebody were utilised for testing. All nine samples were utilised for UCS and CWI testing. Based on these results, four samples were selected for SMC testing. Bond work indices (Ai, BRWi and BBWi) were conducted on all nine samples.

13.2.4.1 Testwork Results

Uniaxial Compressive Strength

The samples tested were classified as hard or very hard. The average strength measured ranged from 152.14 Mpa (KPU104MSM) to 282.22 Mpa for the dolomite rich sample (KPU104SDO2). The testwork results are summarised in Figure 13.7.

Figure 13.7 Summary of UCS Testwork Results



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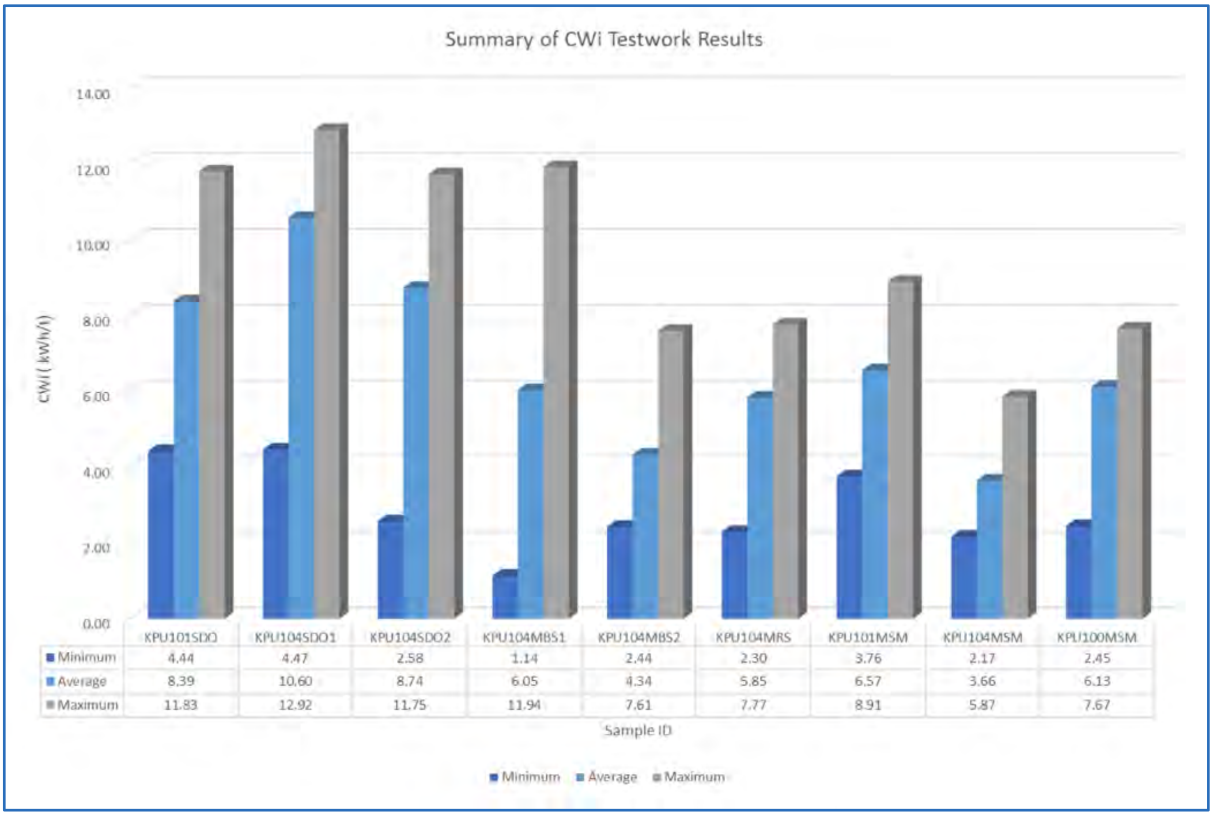
Bond Crusher Work Index

The Bond Crusher Work Index testwork results are summarised in Figure 13.8.

The range of Cwi values are highly variable. Based on the average results, all samples but one were classified within the very soft range (Cwi <10 kWh/t). The exception was KPUSDO1 which has a marginally higher Cwi than other samples tested and was, therefore, classified in the soft range (Cwi from 10–14 kWh/t).

Figure 13.8 Summary of Cwi Testwork Results



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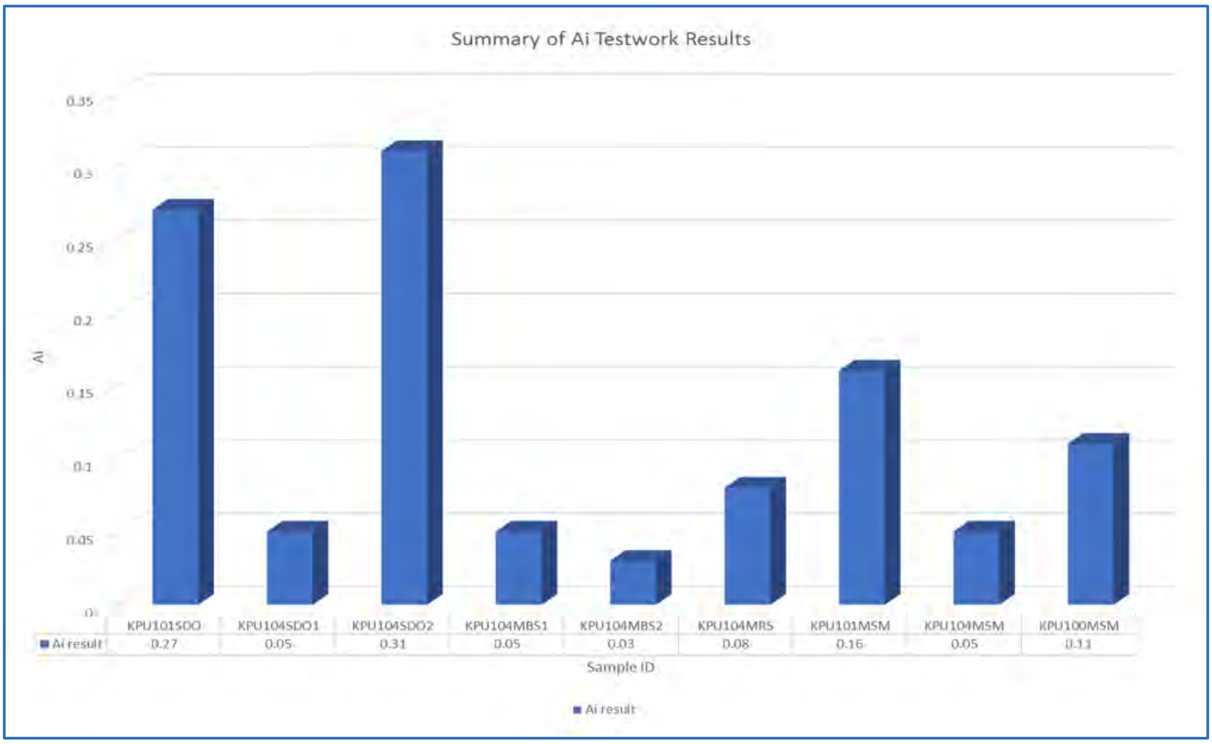
Bond Abrasion Index

The Bond Abrasion Index testwork results are summarised in Figure 13.9.

The abrasion index results indicated that samples ranged from slightly abrasive (0.1–0.4) to non-abrasive (less than 0.1). The measured abrasion index of the samples varied from 0.03–0.30, with only four samples reporting an abrasion index greater than 0.1. It should also be noted that the samples classified as slightly abrasive were derived from the SDO ore type, and the MSM ore types respectively. There is no relationship between abrasiveness and ore type in this small sample set.

Figure 13.9 Bond Abrasion Testwork Results



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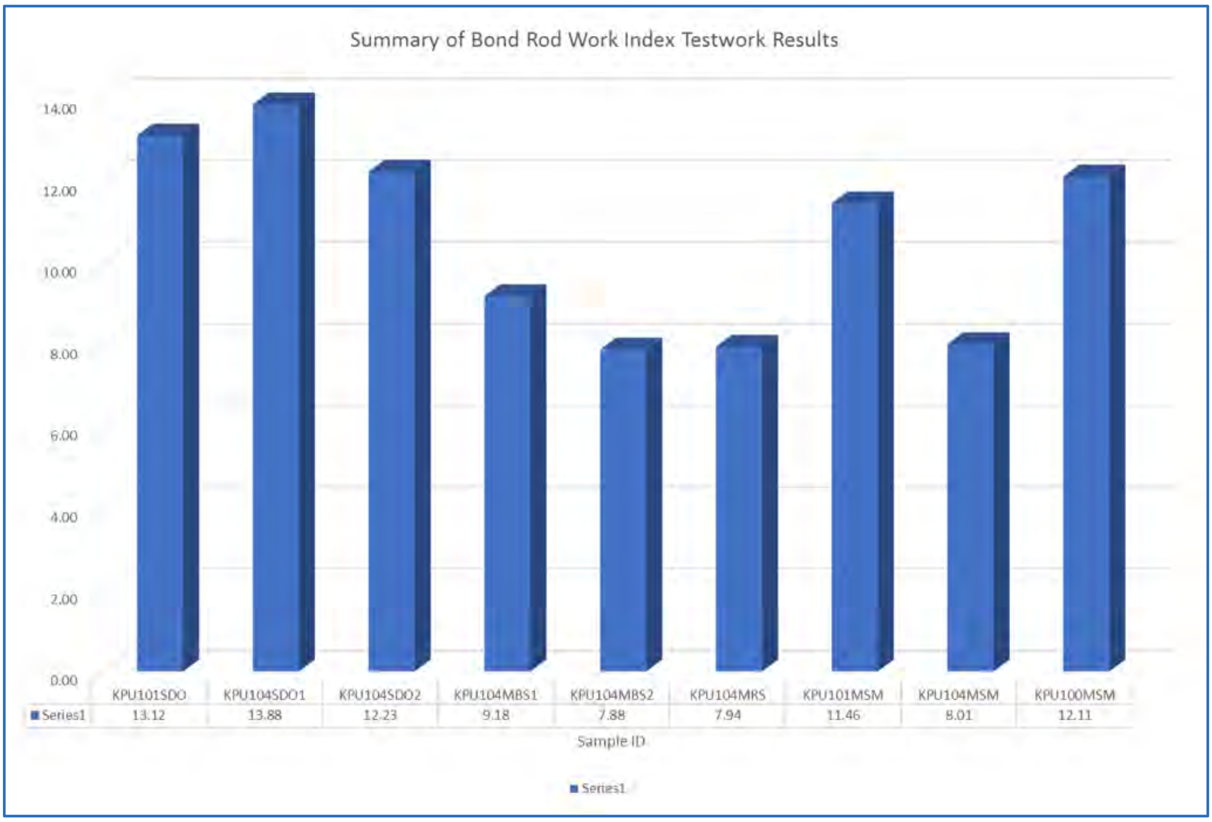
Bond Rod Work Index

The Bond Rod Work Index testwork is summarised in Figure 13.10.

The results indicated that the samples could be classified within the soft to medium hardness ranges with respect to rod milling. The BRWi values ranged from 7.88–13.9 kWh/t and, within the limitations of the small sample set tested, the SDO and MSM ore types are harder than the MBS ore types. The 75th percentile Bond rod work index value was 12.7 kWh/t and the average was 10.7 kWh/t.

Figure 13.10 Bond Rod Work Index Results Summary



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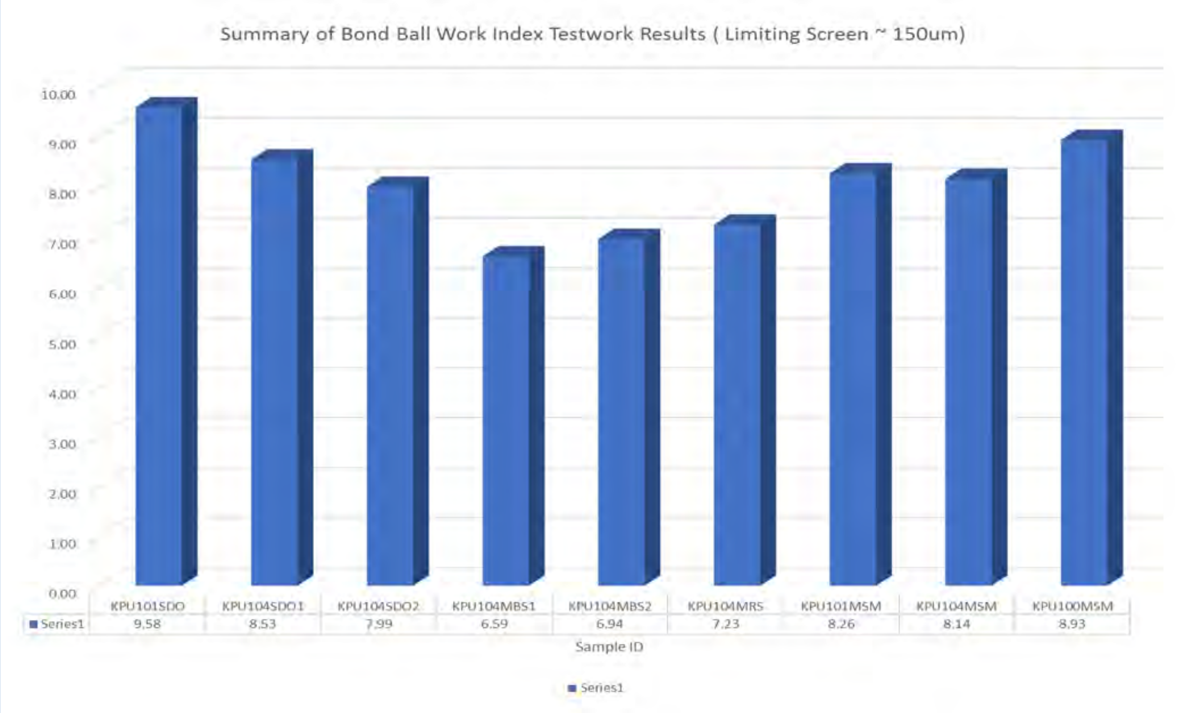
Bond Ball Work Index

The Bond Ball Work Index testwork is summarised in Figure 13.11.

The results obtained indicated that the samples are classified within the soft range with respect to ball milling. The BBWi values ranged from 6.59–9.58 kWh/t. The 75th percentile Bond ball work index value was 8.73 kWh/t with an average of 8.02 kWh/t.

Figure 13.11 Bond Ball Work Index Results Summary



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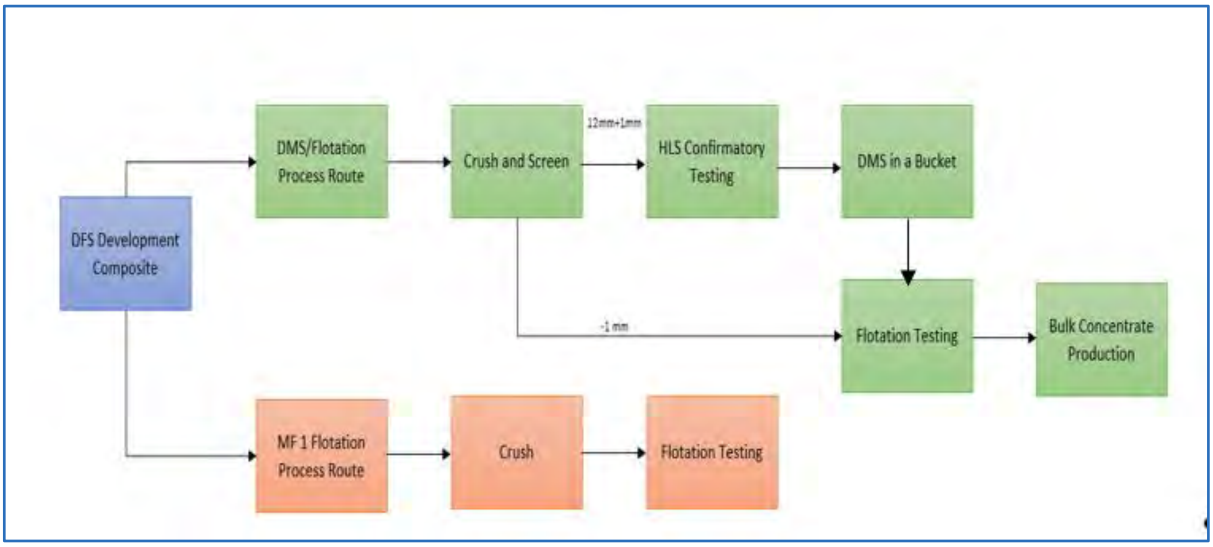
SMC Testing

The SMC testing classified the material as medium hard to very soft relative to the other samples within the JK database. The range of values for Axb was very broad, with the most competent sample reporting an Axb value of 51 (KPU104SDO2) and the least competent sample reporting an Axb value of 142 (KPU104MBS2).

13.2.5 Process Optimisation Testwork

The scope of the optimisation testwork is presented in Figure 13.12.

Figure 13.12 Schematic Overview of Process Optimisation Scope of Work



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The FS development composite was split into two samples with the first being for the standard MF1 (ROM flotation) flow sheet testwork campaign, and the second being for the DMS/flotation flow sheet testwork campaign. The MF1 sample was stage crushed to –1 mm for flotation testing. The sample was split into 1 kg batches and stored in a freezer until required for flotation testing. Flotation scope included baseline testing of both sequential and bulk sulfide flotation, grind optimisation and some reagent optimisation. Once it was established that the DMS/flotation route was to be adopted, focus was diverted to optimisation of this process flow sheet.

The DMS/flotation sample was stage crushed to 100% passing –12 mm and screened to produce –12 mm +1 mm and –1 mm size fractions. The –12 mm +1 mm fraction was subjected to HLS at cut points from 2.7–3.8 t/m^3 in 0.1 increments. Based on the testwork, it was confirmed that the optimal cut point was 3.1 t/m^3, and a bulk sample of DMS concentrate was produced utilising a pilot DMS run with FeSi as media. The DMS concentrate was crushed to –1 mm and was blended with the –1 mm fraction for flotation testing. The sample was split into 1 kg batches and was stored in a freezer until required for flotation testing. The DMS floats produced were prepared and send away for cemented rock fill.

Flotation testing included:

- Baseline testing of both sequential and bulk sulfide flotation,
- Optimal grind testwork,
- Impact of site water on metallurgical performance, and
- Reagent optimisation (largely focussed on pyrite depression).

Once the optimal flow sheet was established, bulk flotation testwork was conducted to generate samples for thickening, filtration, and environmental studies.

It should be noted that all testwork samples were stored in a freezer to preserve their condition once crushed.

13.2.5.1 Gravity Separation Testwork

Confirmatory HLS testwork was conducted on three separate sets of samples for reproducibility.

The outcome of the HLS testing on the –12 mm +1 mm fraction confirmed the outcomes of testwork in the PFS phase:

- Mass rejection of approximately 30% could be achieved at a cut point of 3.1 t/m^3.
- At this cut point, Zn recovery to the sinks fraction was as high as 98.94%.
- Changing the top size of the DMS feed from 20 mm to 12 mm had a negligible impact on overall Zn recovery while making the DMS feed more suited to pumping and cyclone DMS separation. A sinks top size of 12 mm is also suited to being fed to a full scale ball mill without further crushing.

13.2.5.2 Flotation Testwork on ROM Samples

Optimal Grind Testing

Flotation testwork was conducted at three grinds (80% passing 75 µm, 106 µm, and 150 µm) and was initially conducted utilising sequential flotation conditions defined for the PFS study. The testwork was conducted as a rougher kinetic test. Based on the grade and recovery relationship, it was noted that a grind of 80% passing 106 µm produced metallurgically similar results to the finer grind of 80% passing 75 µm. It was also noted that Zn losses to the Cu-Pb concentrate increased as the grind size was reduced, and as such, the coarser grind was favoured. Reducing the grind from 80% passing 106 µm to 80% passing 75 µm increased Zn losses to the Cu-Pb circuit by 4.8%.

At a grind of 80% passing 75 µm, the Zn content could be upgraded from 32.4–55.4% Zn, with a final Zn recovery to concentrate of 95.9%. In addition to this, approximately 56.4% of Fe present in the feed reported to tailings, of which, 15% was rejected in the Cu-Pb flotation stage.

Reagent and Flow sheet Optimisation Studies

The major focus of testwork was to develop a process flotation route for ROM ore that was robust enough to cater for ore with increased Cu-Pb content, as well as variable pyrite content. Three possible flotation scenarios were investigated:

- Selective flotation, utilising standard NaCN depression reagent regime,
- Bulk sulfide flotation, and
- Selective flotation, using alternative non-cyanide-based regimes.

The programme was brief, as midway through the programme, the decision was made to proceed with the flow sheet where the ore would be pre-concentrated by DMS ahead of flotation. The scope of the testing for each scenario and outcomes are as follows:

1. Selective Flotation Route – Cyanide Based:

   The testwork explored variations in reagent dosages compared to the baseline conditions defined during the PFS. A higher dosage of NaCN/$ZnSO_4$ blend (400 g/t and 800 g/t respectively) was utilised to pre-float the Cu-Pb ahead of Zn flotation.

2. Selective Flotation Route – Non-Cyanide Based:

   The use of highly selective Cu-Pb collectors (which are claimed to have excellent selectivity against pyrite) were tested in this series of work. The reagents and rates tested were Aero 3418A (15 g/t) and MX6206 (15 g/t, 10 g/t, and 3 g/t). A final test was conducted where lime was substituted for soda ash as pH modifier in the sequential flotation regime.

   It should be noted that the tests where MX6206 was used as collector at low dosages had very high mass pulls to the Cu-Pb concentrate and were deemed to have failed due to obvious high loss of zinc. There was no requirement to chemically analyse the products nor include the results in any further evaluations.

3. Bulk sulfide flotation route:

   The baseline bulk sulfide flotation reagent regime defined in the PFS stage was repeated and this was followed by two tests utilising the potential pyrite depressants Pionera F250 and Aero 7261A.

   None of the non-cyanide based sequential reagent systems tested could produce better metallurgical performance than the baseline test. It was noted that increasing the depressant dosage produced superior performance.

It was also noted that the bulk sulfide testwork could not produce a high-grade final Zn concentrate, and that this could not be improved by using the proposed pyrite depressants. The Zn concentrate grade could not be improved by making use of an additional cleaner stage. The bulk flotation regime also failed to improve upon the Zn recovery obtained by the baseline cyanide based sequential flotation flow sheet.

13.2.5.3 Flotation Testwork on Preconcentrated Samples

Optimal Grind Testing

For this phase of testing, it was assumed that the optimal grind would be as per testwork conducted on the ROM samples, and therefore, a grind size of 80% passing 106 µm was adopted for all tests conducted.

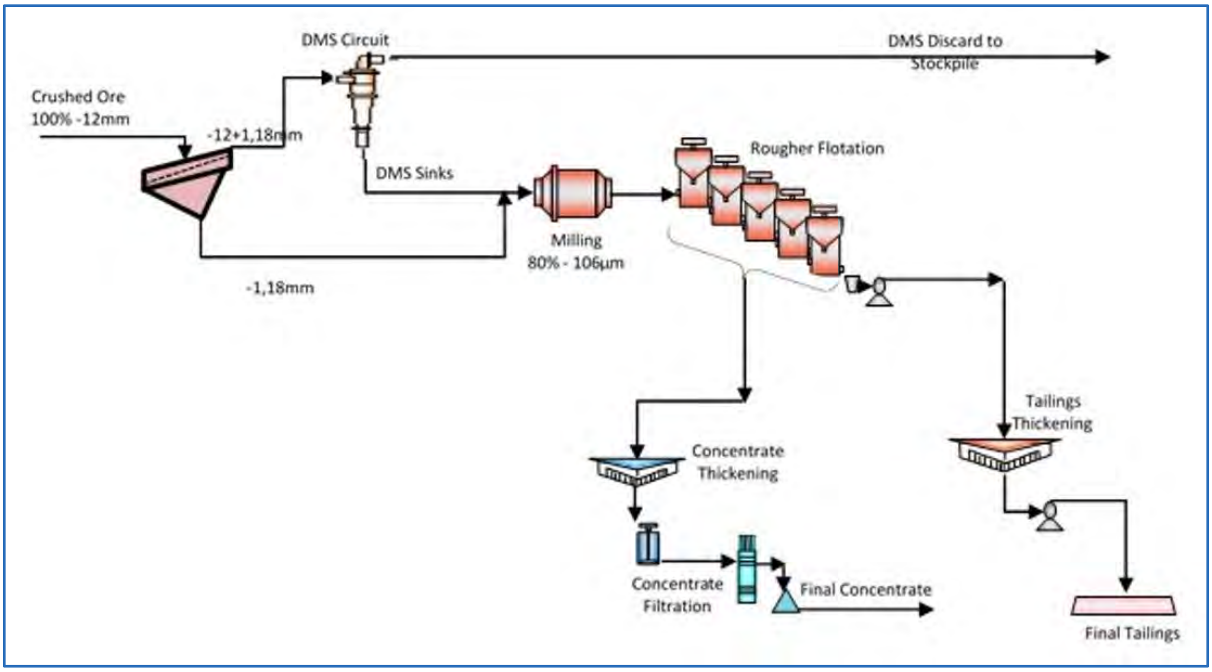
Baseline testing of both the sequential and bulk sulfide flotation circuits yielded the following results:

- Sequential flotation under conditions defined for the PFS yielded a final Zn concentrate of 49.7% Zn, at an overall recovery of 95.4% Zn.
- Bulk flotation under conditions defined in the PFS yielded a final Zn concentrate of 54.8% Zn at a Zn recovery of 96.4% Zn.

Process Development and Reagent Optimisation

A number of flotation tests were conducted using the pre-concentrated DMS sinks and natural fines as feed. The feed sample was milled to an optimum grind size of 80% passing 106 µm and subjected to various flotation conditions. Bulk sulfides flotation circuit illustrated in Figure 13.13 achieved the best results with zinc recovery of 96.4% at a concentrate grade of 54.8% Zn.

Figure 13.13 Schematic Overview of Kipushi FS Process Flow sheet



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The circuit will consist of crushing the material to a product of 100% –12 mm, this product is then screened using a 1.18 mm aperture size; –12 +1.18 mm is conveyed to the DMS plant and the fines at –1.18 mm is combined with the DMS concentrate to milling where a product size of 80% passing 106 µm is targeted. Ground product is then conditioned with reagents for bulk sulfide float, the flotation concentrate will be thickened, filtered, and packaged for dispatch to the market.

DMS tails will be conveyed into a stockpile for use in back fill and flotation tails will be thickened and pumped to the tailing storage facility. The LOM average head sample assaying ~32% Zn achieved overall recovery of ~95% at a concentrate grade of 54.7% Zn. This circuit has been adopted for the FS, however, precautionary measures are being considered to ensure that concentrate grade ≥53.5% is consistently achieved even in periods when material from areas with higher than average base metal sulfides (esp. pyrite) composition will be processed.

Bulk sulfides test results for the various conditions tested are presented in Table 13.7. It should be noted that the tests where MX6206 was used as collector at low dosages had very high mass pulls to the Cu-Pb concentrate and were deemed to have failed due to obvious high loss of zinc. There was no requirement to chemically analyse the products nor include the results in any further evaluations.

Table 13.7 Summary of Outcomes of Reagent Optimisation Testing at 10 min Float

Test	Reagent Regime				Testwork Outcomes				
	Lime (g/t)	$CuSO_4$ (g/t)	SIPX (g/t)	MIBC (g/t)	Mass Pull (%)	Zn (%)	S (%)	Fe (%)	Zn Recovery (%)
Test 5 – Aero 7261A plus aeration	800	900	80	36	85.30	48.70	35.90	9.90	99.40
Test 6a	800	900	40	36	74.30	57.80	34.00	7.69	96.40
Test 6 – Bulk	800	900	40	36	77	53.40	35.50	8.40	95.90
Test 6c	800	900	40	36	77.50	51.90	33.80	10.00	98.70
Test 11 – Site water	2,400	900	40	36	65	54.70	34.00	6.24	85.50
Test 13 – Low % solids	2,469	900	40	48	76.50	53.70	32.70	7.93	95.90
Test 14 – Reduced SIPX	1,020	900	30	36	75.40	52.20	35.20	7.67	93.90
Test 15 – Reduced SIPX – Stagewise	810	900	30	36	79.60	50.80	34.90	8.70	95.60
Test 16 – SMBS at 1,000 g/t	0	900	40	36	14.90	No Assay – mass pull too low			
Test 17 – Aero 500 at 8 g/t and Stagewise SIPX	800	300	42	36	71.20	53.40	41.40	7.40	90.40
Test 18 – Aero 500 at 8 g/t and reduced SIPX	800	900	32	36	80.70	49.90	37.50	9.01	95.50
Test 19 – Aero 500 at 8 g/t and reduced SIPX	800	600	32	36	80.80	50.90	36.90	8.56	96.70
Test 20 – reduced SMBS	0	900	40	36	18.10	No Assay – mass pull too low			
Test 21 – 170 g/t Cytec7261 – Fe Control	800	600	40	36	23.20				

Effect of Site Water on Metallurgical Performance

Samples of water from the Kipushi site were furnished for testwork. Two tests were conducted utilising site water, and it was noted that the lime addition to these tests was significantly higher than the baseline conditions. Lime addition to these tests was as high as 2,400 g/t compared to the baseline condition of approximately 800 g/t, indicating that the site water is buffering pH adjustment. Water derived from the Kipushi site is known to be particularly hard, ranging from 590–788 mg/l $CaCO_3$, and this would explain this effect.

In addition to this, it was noted that the use of site water impacted significantly on overall recovery, with a final concentrate grade of 54% Zn attainable, but at recoveries as low as 86.5%.

For the project, the decision was made to utilise potable water for make-up. The tailings return water is also not returned to the process circuit. An exhaustive analysis of the effect of water quality was not conducted, and it is reasonable that there would be an acceptable water quality somewhere between the site water and potable water which will reduce water treatment requirements.

13.2.6 Variability Testwork

The nine variability samples were crushed and screened into –12 mm +1 mm and –1 mm fractions respectively. The –12 mm +1 mm fractions were characterised using HLS testing at a single cut point (3.1 t/m^3) to establish perfect metallurgical performance in DMS. A second sub-sample of the –12 mm +1 mm material was pre-concentrated utilising the more realistic 'DMS in a bucket' technique at a cut-point of 3.1 t/m^3 and was re-combined with the –1 mm fraction in order to produce a mill feed for flotation. The samples were then stage crushed to –1 mm for flotation testing.

Milling curves were conducted for each of the nine samples, to establish milling time required to produce a grind size of 80% passing 106 µm. These milling times were utilised for variability flotation work.

Each of the nine flotation samples were subjected to bulk sulfide flotation under the conditions presented in Table 13.8.

Table 13.8 Baseline Flotation Conditions for Variability Samples

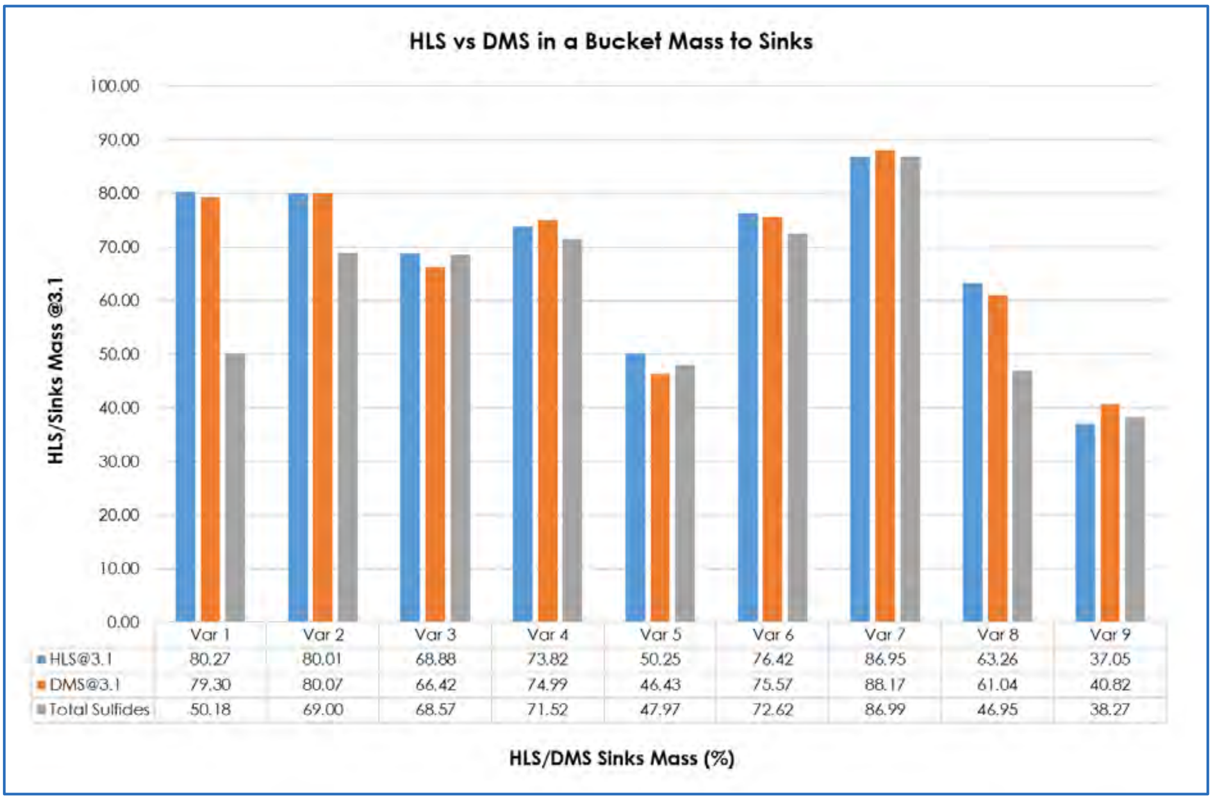
Description	Unit	Value
Flotation Time		
Rougher 1	Minimum	2
Rougher 2	Minimum	4
Rougher 3	Minimum	4
Rougher 4	Minimum	2
Rougher 5	Minimum	2
Total	Minimum	14
Reagent Dosage		
Lime		As required, to maintain pH of 11.5
$CuSO_4$	g/t	900
SIPX	g/t	40
MIBC	g/t	46

13.2.6.1 Gravity Separation Testwork

The variability sample results of the HLS and 'DMS in a bucket' tests (both at a nominal cut point of 3.1 t/m^3) are compared in Figure 13.14.

Good correlation was noted between the total sulfide content within the orebody (based on mineralogical evaluation) and the mass of material reporting to the sinks fraction, exceptions to this were variability sample 1 and variability sample 8.

Figure 13.14 Correlation between DMS/HLS mass pull and Feed Sulfide Content



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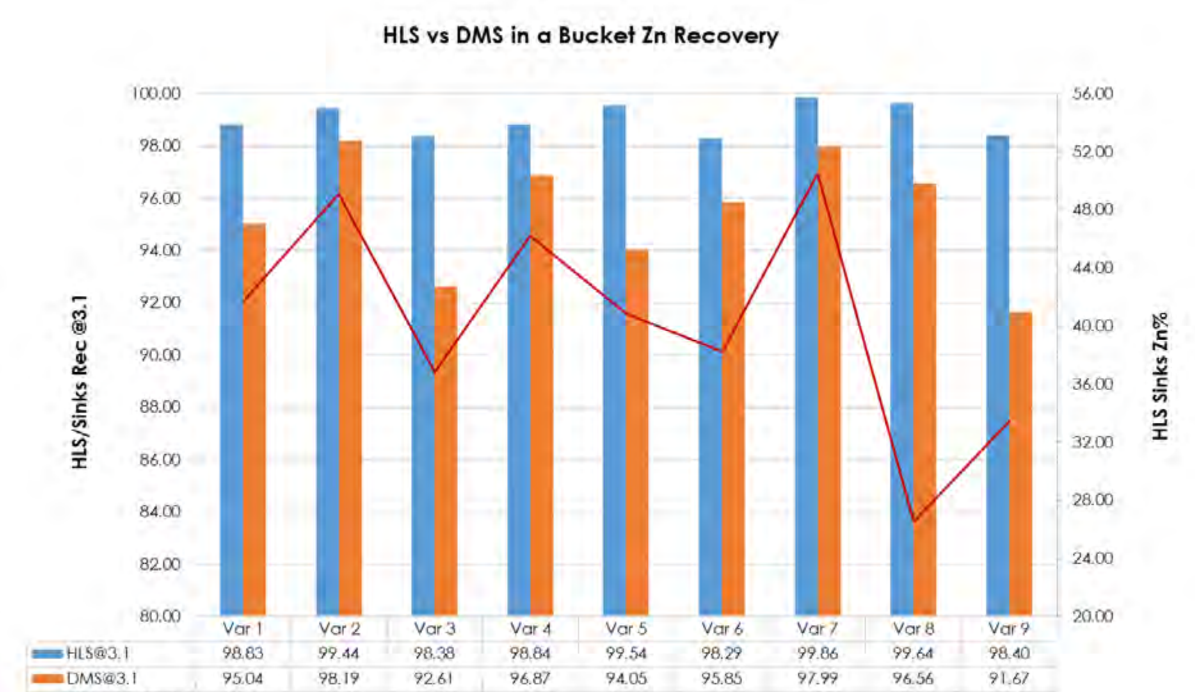
Zn recovery to HLS and DMS sinks are presented in Figure 13.15. Based on the presented figures, Zn recoveries for HLS testing ranged from 98.3–99.9%. Recoveries reduced for the corresponding DMS separations on the same samples, where the range was from 91.7–98.2%. HLS separation can be considered perfect and theoretical while ‘DMS in a Bucket’ provides a more realistic outcome for design purposes. It should be noted that the ‘DMS in a bucket’ technique is far less efficient than a DMS cyclone, where it was previously demonstrated during testwork that recoveries of up to 99% were attainable.

As expected, the DMS recovery is consistent between the samples tested, and is independent of the Zn head grade at the proposed cut point. The mass pull to sinks is largely dependent on the proportion of sulfides present within the feed. As such, the degree to which zinc can be upgraded is dependent on the proportion of sulfides present in the feed (which correlates to mass pull), and the proportion of sphalerite in the mix of sulfides.

Sinks concentrate grades ranged from 27–51% Zn, but the 27% result (together with the 33% result) is very low and unlikely to occur as it would require all DMS feed to be close to mine cut-off grade. A more reasonable expected range for DMS concentrate grade is from 38–51% Zn.

'DMS in a bucket' recoveries were lower than expected from the HLS testing, but recoveries are expected to be higher in a real DMS plant. Media conditions are difficult to control in a bucket separation compared to a well-run DMS cyclone plant and a more consistent separation is achieved due to the high G-forces achievable in the cyclone.

Figure 13.15 Zn Recovery and Grade for HLS and DMS Sinks at a Cut Point of 3.1 t/m^3



	Var 1	Var 2	Var 3	Var 4	Var 5	Var 6	Var 7	Var 8	Var 9
HLS@3.1	98.83	99.44	98.38	98.84	99.54	98.29	99.86	99.64	98.40
DMS@3.1	95.04	98.19	92.61	96.87	94.05	95.85	97.99	96.56	91.67
HLS Sinks Grade	41.70	49.10	36.84	46.24	40.85	38.23	50.50	26.56	33.42

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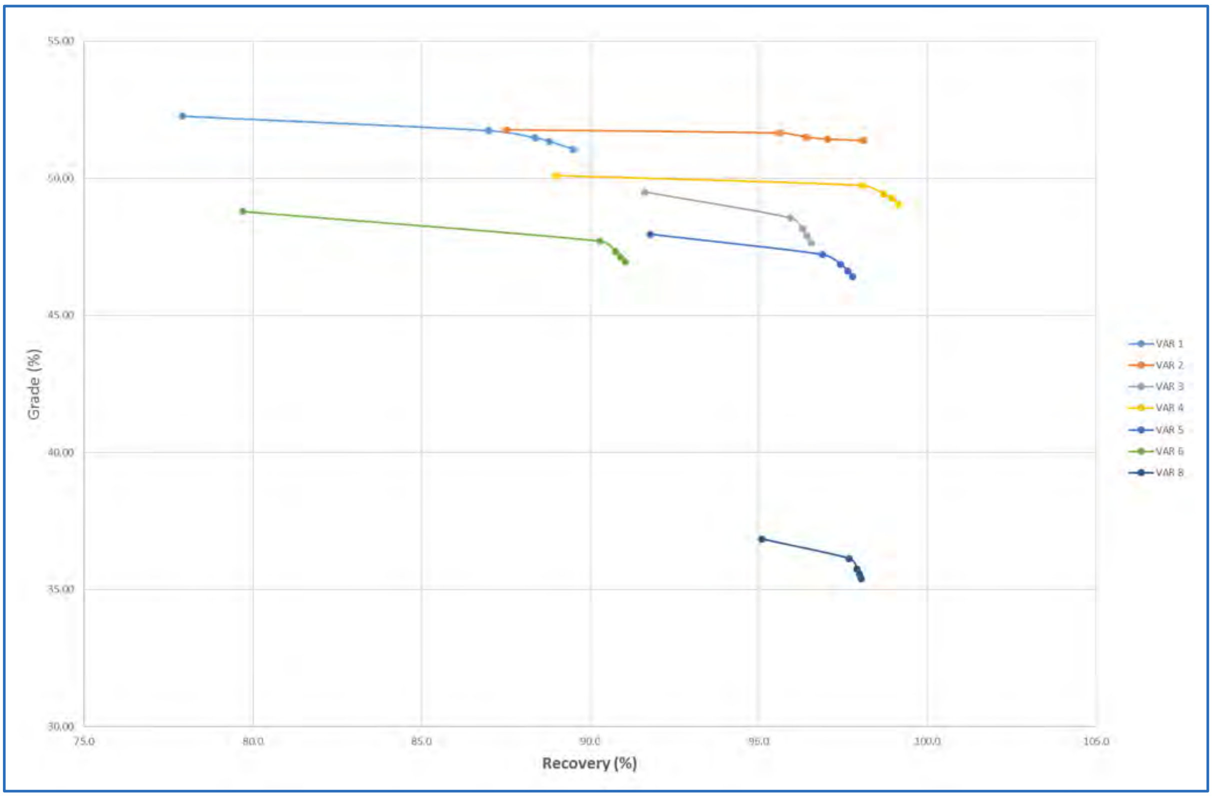
13.2.6.2 Flotation Testwork

Flotation tests were also conducted using the test conditions that achieved the optimum results with the FS sample. The outcomes of flotation testing on the variability samples after upgrading the coarse fraction by means of 'DMS in a bucket' are presented in Figure 13.16.

Figure 13.16 Flotation Metallurgical Performance for Nine Variability Composites



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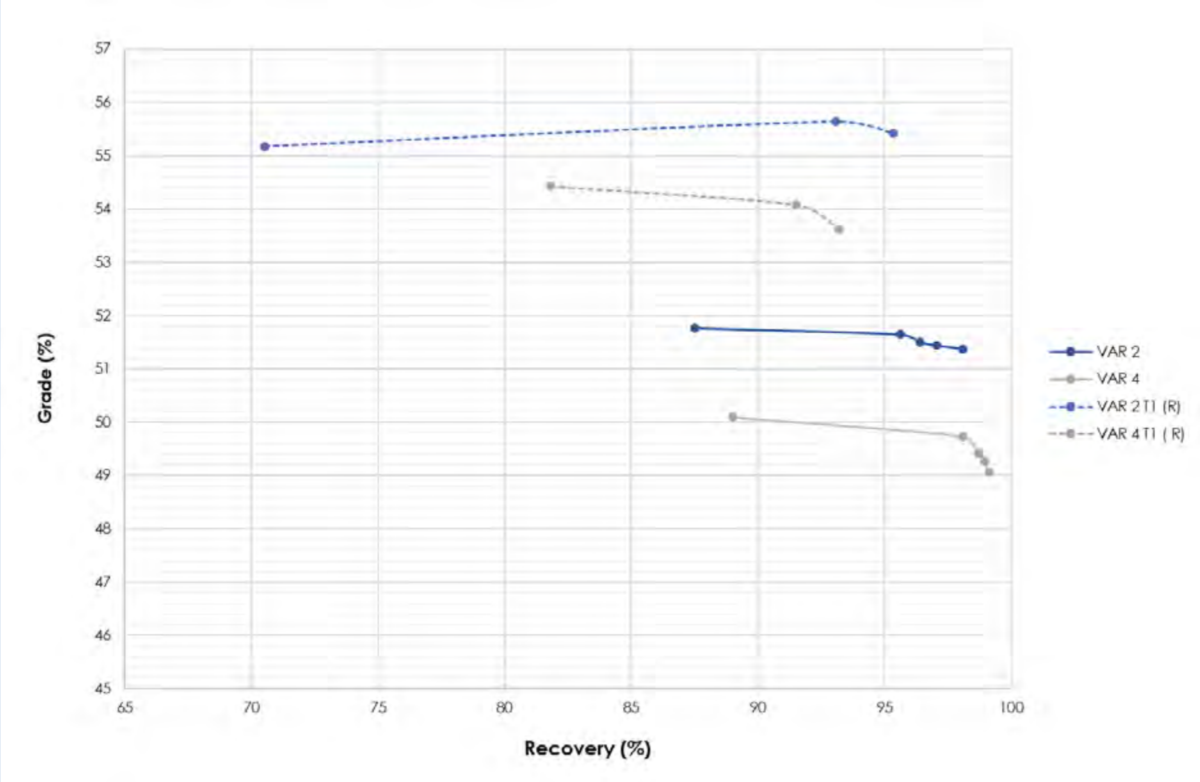
For all variability concentrates except Var 1, flotation Zn recovery was >90%. Var 1 Zn recovery was just below 90%. It was noted that only two of the samples tested were not able to attain the required concentrate grades above 50% Zn. During the laboratory scale flotation process, it was observed that the froth production rates were very rapid and would flow uncontrollably over the lip of the cell. As a result, pulp was carrying over the lip with the froth and mass pulls were attained which were well over than anticipated.

It was decided to conduct repeat flotation testing where samples were still available (Variability 2 and Variability 4). In these tests, the air flow rates into the pulp levels were reduced marginally and the air flow rates into flotation cell were as carefully controlled to regulate the froth flow over the lip of the cell. It was observed that mass pulls were significantly reduced in this manner, and that these samples would be upgraded to target utilising this philosophy, as is presented in Figure 13.17.

Figure 13.17 Outcome of Repeated Flotation Testing of Selected Variability Samples



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It can, therefore, be concluded that the original tests were performed under conditions that were un-optimised and that the majority of the variability samples tested should be able to exceed 50% Zn concentrate grade, provided that the froth flow rates, and mass pulls are carefully controlled within the flotation circuit. Reiterating the previous observation that all variability samples had average or higher iron grades, the flotation circuit is expected to produce concentrates in the range 53–57% Zn.

13.2.7 Equipment (Vendor) Testwork

13.2.7.1 Concentrate Thickening Testwork

A bulk sample of flotation concentrate was supplied to Outotec and to Tenova Delkor to conduct testwork required for sizing of the concentrate thickener, as well as most appropriate flocculant and optimal flocculant dose. The scope of work was designed to optimise and establish the following for design purposes:

- Flocculant selection and optimal dosage,
- Optimal feed dilution,
- Solids loading,
- Overflow clarity,

- Achievable underflow density, and
- Underflow rheology.

Outotec used a 99 mm Supaflo high rate thickener test rig and Tenova Delkor used their dynamic rig.

The testwork conducted by Outotec indicated that the Zn concentrate was readily thickened to solids concentrations of up to 75% (w/w) and would produce a very clear thickener overflow of less than 100 mg/l TSS, at a solids flux of 0.25 t/m^{2}h. Optimum feedwell dilution was established to be 17% solids. The Outotec testing was conducted utilising the Superfloc suite of flocculants, and the most amenable flocculant was Supafloc N150, at a dosage rate of 20 g/t.

Testwork conducted by Tenova utilised Magnafloc products for flocculant screening, and recommended the use of Magnafloc 1011 for both concentrate and tailings thickening, at a dosage of 15 g/t. This was utilised as a common flocculant for the FS campaign. The results of testwork conducted by Tenova were consistent with those obtained from Outotec.

13.2.7.2 Concentrate Filtration Testwork

The objective of the Outotec testwork was to establish filtration rates, final moisture contents and filter cake thickness for tower filtration on pre-thickened bulk sample of Zn concentrate prepared for the Kipushi Project. The outcomes were as follows:

- Total cycle time ~11.5–14.5 min.
- Filter cake moisture content ~8.8–10.2% (w/w).
- Filter cake thickness ~37–42 mm.
- Filtration rate ~406–545 kg/m^{2}h.

13.2.7.3 Tailings Thickening Testwork

A bulk sample of flotation tailings was supplied to Tenova Delkor to conduct testwork required for sizing of the tailing's thickener, as well as the most appropriate flocculant and its optimal dose. The scope of work was designed to optimise and establish the following for design purposes:

- Flocculant selection and optimal dosage,
- Optimal feed dilution,
- Solids loading,
- Overflow clarity,
- Achievable underflow density, and
- Underflow rheology.

A dynamic test rig was utilised to conduct this testwork, furnished, and operated by Tenova Delkor.

The testwork conducted indicated that the tailings sample tested was readily thickened to solids concentrations in excess of 87% (w/w) and would produce a very clear thickener overflow of less than 100 mg/l TSS, at a solids flux of 0.34 t/m²h. Optimum feedwell dilution was established to be 20% solids.

Tenova utilised Magnafloc products for flocculant screening, and recommended the use of Magnafloc 1011 for both concentrate and tailings thickening, at a dosage of 15 g/t. This was utilised as a common flocculant for the Kipushi 2022 FS campaign.

13.2.7.4 Pre-flotation Thickening Testwork

No testwork was conducted on this sample. It was assumed that the material would conservatively require a solids flux of 0.34 t/m²h, and this was applied to the design.

Concentrate Quality and Deleterious Elements

Estimated concentrate quality in terms of chemical composition, and associated penalties for off-specification products was derived from the chemical analysis of bulk concentrates produced during the testwork campaign from the FS master composite. The results showed that under average ROM conditions, the concentrate meets the requirements in terms of deleterious elements. However, the Fe and Mg grades of this concentrate are within close range of the prescribed limits for these elements, and as such, penalties could be incurred on occasions. In addition to this, it should be noted that, because a bulk sulfide concentrate is produced, the concentrate grade will be sensitive to Cu and Pb feed grades, as well as pyrite content.

Sphalerite is the dominant mineral, comprising 78.3% of the final concentrate. The results indicated that there are no hazardous silicates present in the concentrate, and that the largest gangue phase present is pyrite, followed by minor amounts of dolomite, galena, and chalcopyrite.

The expected Fe content of the concentrate could exceed the prescribed 8% limit. The deportment and nature of Fe present in the final concentrate should be well understood to optimise concentrate revenue. Further studies were conducted to establish the deportment of the various elements in the mineral phases of the concentrate (EMPA analysis). 81% of the Fe present in the concentrate sample was contained in pyrite, with minor amounts present in non-sulfide gangue. There is potential to reduce concentrate Fe grade if pyrite recovery to the flotation concentrate can be further reduced.

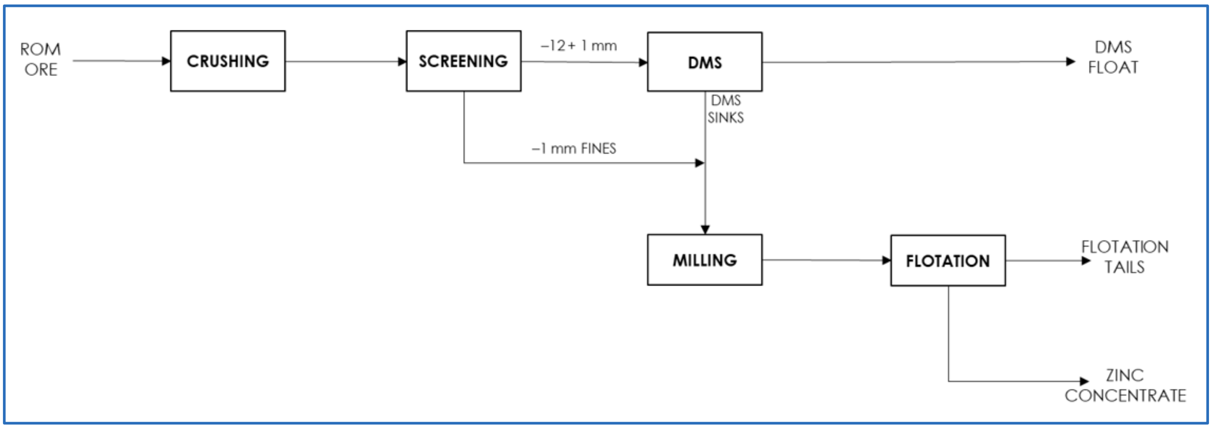
Pyrite is predominantly within the 20–50 µm size range, and a significant proportion of pyrite present in the concentrate is poorly liberated or locked. In terms of mineral association, approximately 70% of pyrite in the Zn concentrate occurs as free surface (liberated with respect to flotation response), and a further 20% of pyrite in the concentrate is associated with sphalerite. Other pyrite associations are minor. This suggests that pyrite content within the concentrate could be reduced by improved cleaning and targeting rejection of the 70% of pyrite that is liberated.

13.3 Recovery Estimation

The Kipushi Zinc project involves a new process plant design, based on the testwork results as presented in sections of this report. The overall process mineral recovery estimation is based on the process flow sheet involving crushing and screening, milling and flotation, as presented in Figure 13.18 below.

The overall process plant recovery parameters are derived from testwork data analysis and appropriate recovery model simulations. For the KICO process plant, the overall mineral recovery is determined from the component circuit recoveries making up the final concentrate product as per the flow sheet.

Figure 13.18 Process Flow Sheet



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13.3.1 Overall Process Recovery Model

The flotation variability data obtained was utilised to establish a mass pull vs recovery model for flotation performance. The model utilises simple flotation kinetics, fitted to the variability data and FS development sample data.

The equation fitted is as follows:

$$Rec = Rmax(1 - \left(1 - \frac{MP}{100}\right))^{\wedge}b$$

Where:

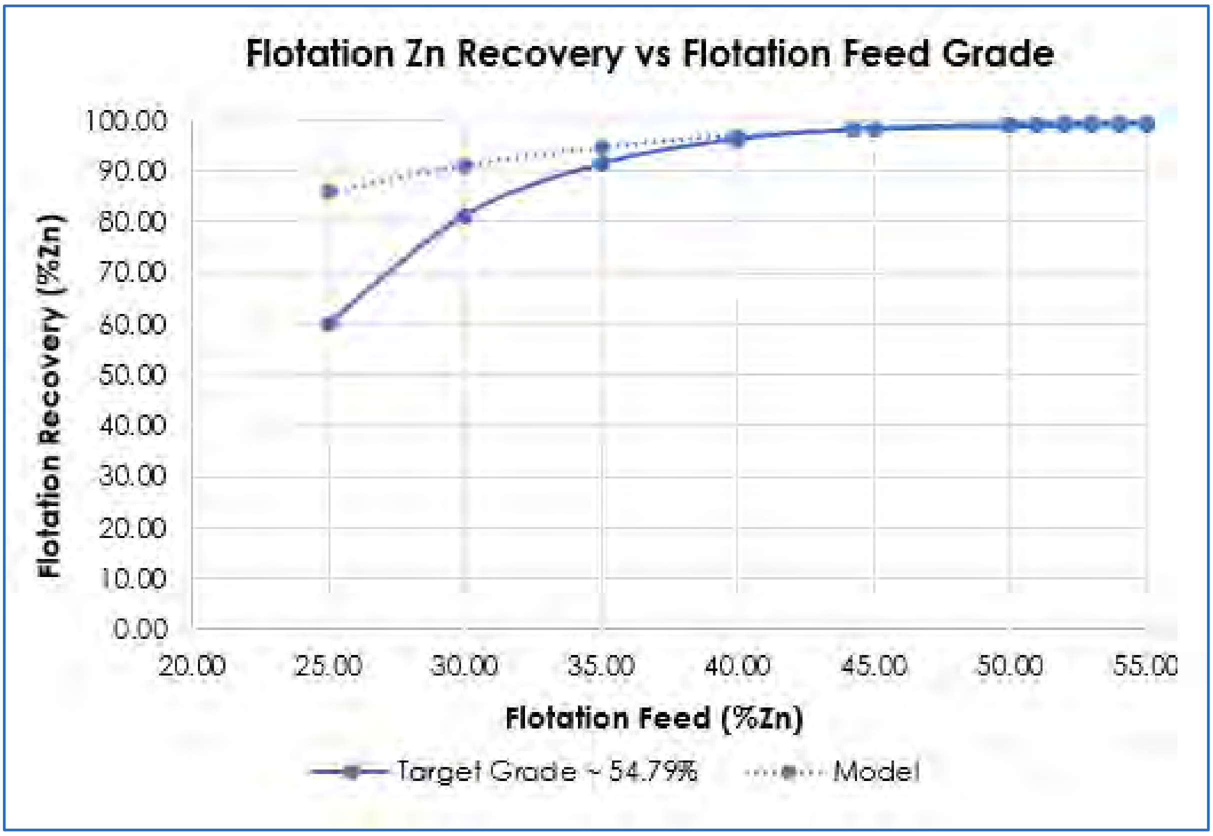
- Rmax is the maximum Zn recovery attainable, based on the variability testwork.
- MP is the mass pull to concentrate, and
- b is a coefficient determined by curve fitting.

The methodology utilised to best fit the parameters to the data was to adjust these parameters to produce a minimum sum of squares of the difference between the measured values and the model outputs. Three data points were excluded, as per client request.

The flotation model was then utilised to produce a head grade vs recovery relationship for a target concentrate grade of 54.8% Zn, as presented in Figure 13.19.

This model was then utilised for establishing the overall recovery model.

Figure 13.19 Flotation Zn Recovery as a Function of Head Grade at a Target Zn Concentrate grade of 54.79% Zn



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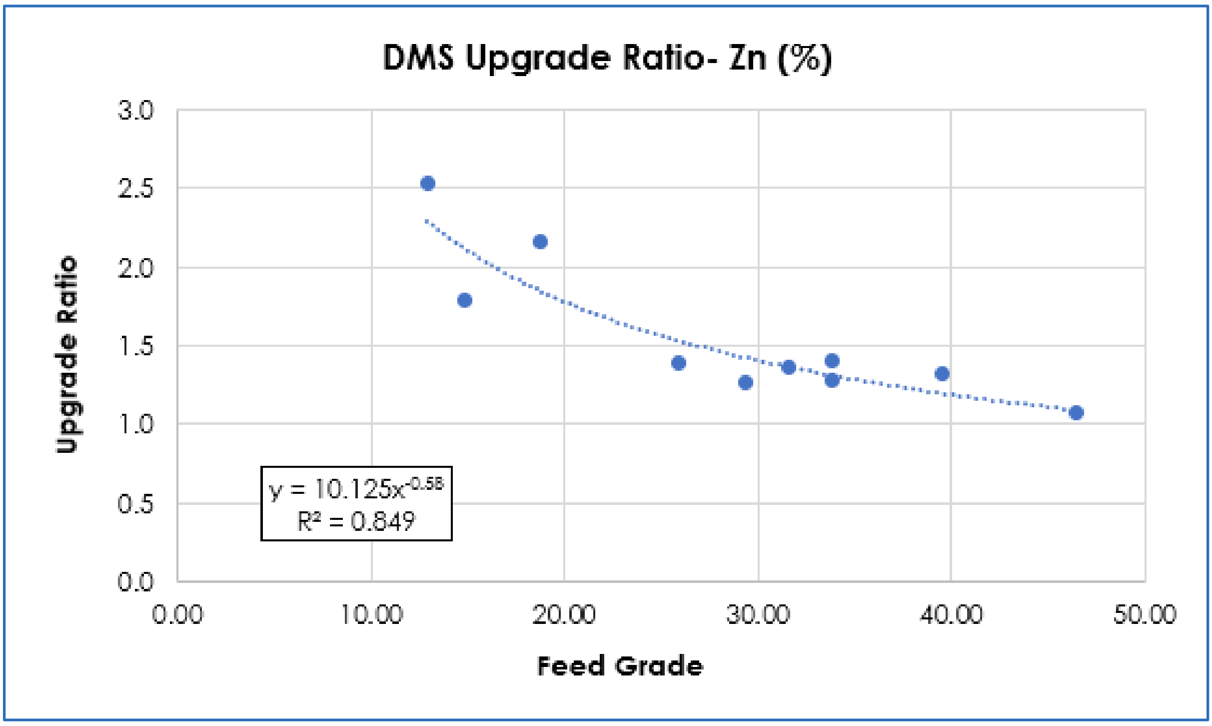
The overall recovery model was built by producing mass and metal balances for a range of Zn head grades to obtain a concentrate grade of 54.8% Zn. The produced overall Zn recovery as a function of head grade was fitted to obtain the overall head grade and recovery relationship for the project. The model had several inputs:

- The proportion of fine material in the crushed ore product was calculated from crushing simulations (JKTech and Sandvik). The mass pull to the fines fraction was therefore fixed for all head grades at 26%.

- It was assumed that no preferential upgrading occurs between the –12 mm +1 mm fraction and –1 mm fraction. For the overall model, the head grade of the ROM feed was therefore applied to both the –12 mm +1 mm fraction and the –1 mm fraction.
- The DMS concentrate upgrade was calculated utilising a correlation between DMS feed grade and the upgrade ratio from DMS feed grade to concentrate grade, as is presented in Figure 13.20.
- It was noted that the 'DMS in a bucket' process utilised for upgrading the variability samples was not as efficient as pilot scale DMS cyclone testing on the FS development composite. For this reason, DMS recovery was fixed at 99%, based on the results attained on pilot scale DMS testing, as well as the DMS simulation packages.
- The flotation model presented in this section was applied to the mill feed grade, assuming a target concentrate grade of 54.8% Zn.

Figure 13.20 DMS Concentrate Upgrade Ratio



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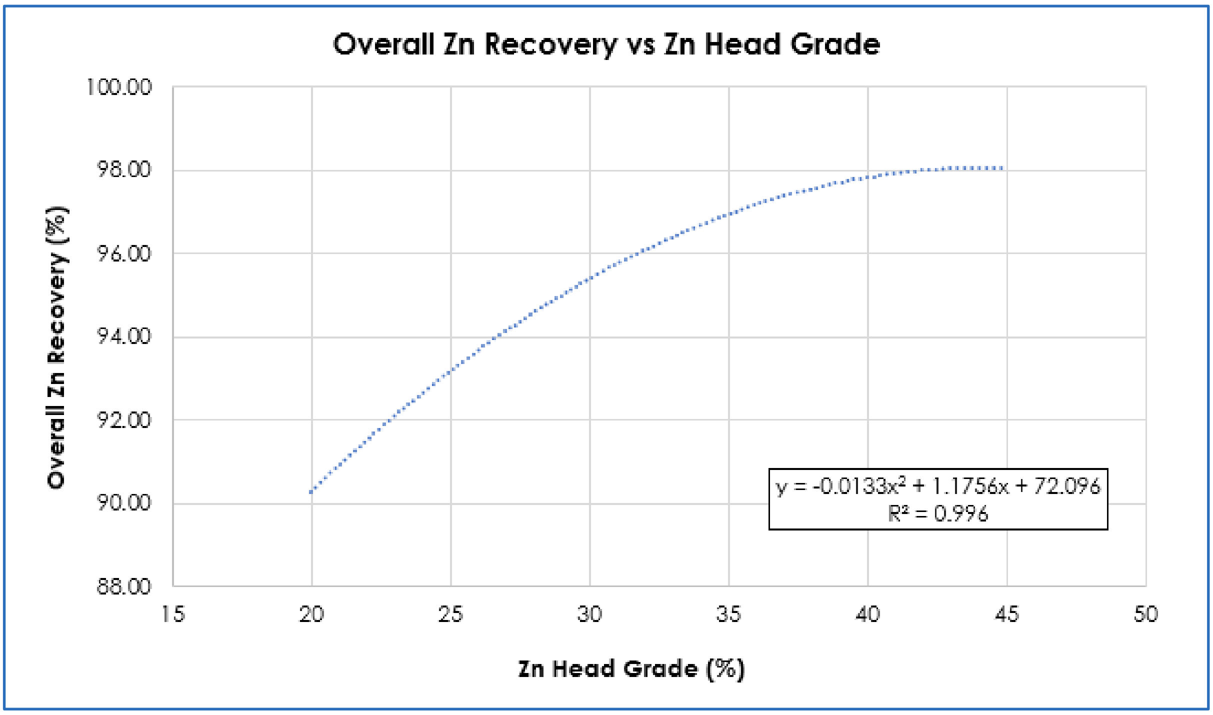
The recovery formula for the project, illustrating the relationship between Zn ROM head grade and the overall Zn recovery is presented in Figure 13.21.

Figure 13.21 Relationship between Zn Head Grade and Overall Zn Recovery



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13.4 QP Comments on Section 13

The testwork performed for the FS covered all aspects of the proposed circuit and, with some interpretation, has confirmed the flow sheet. The main interpretation requirements, which in the opinion of the QP are reasonable, are:

- Metallurgical testwork were completed on representative samples from the planned mining area, and that the test results evidently proved that a saleable flotation concentrate specification could be attained.
- The testwork resulted in exceptionally good DMS separation and dolomite waste discard ability. However, it was also confirmed that the process circuit with DMS alone would be unable to generate consistent saleable product. This is due to the concentrate grade dilution by the presence of medium-high content levels of pyrite in the ore, which reports to the DMS concentrate.
- That the selection of flotation technology and design of the flotation circuit will be critical to the attainment and control of design concentrate grades
- The metallurgy is assisted significantly by the globally high Zn head grades.

14 MINERAL RESOURCE ESTIMATES

This section has not been changed from the Kipushi 2019 Resource Update and remains the most current study work available.

On behalf of the Kipushi Corporation SA (KICO), the MSA Group Pty Ltd (MSA) has completed a Mineral Resource estimate for the Kipushi Project (Kipushi).

To the best of the Qualified Person's (QPs) knowledge there are currently no title, legal, taxation, marketing, permitting, socio-economic, or other relevant issues that may materially affect the Mineral Resource, aside from those already mentioned in Section 4 of this report.

The Mineral Resource estimate incorporates drilling data collected by KICO from March 2014 until November 2015 inclusive and May 2017 to November 2017 inclusive, which, in the QP opinion, were collected in accordance with The Canadian Institute of Mining, Metallurgy and Petroleum (CIM) 'Exploration Best Practices Guidelines'. Previous drilling work completed by La Générale des Carrières et des Mines (Gécamines) has been incorporated into the estimate following the results of a twin drilling exercise and verification sampling of a number of cores.

The Mineral Resource was estimated using the 2003 CIM 'Best Practice Guidelines for Estimation of Mineral Resources and Mineral Reserves' and classified in accordance with the '2014 CIM Definition Standards'. It should be noted that Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

The Mineral Resource estimate was conducted using Datamine Studio RM software, together with Microsoft Excel, JMP, and Snowden Supervisor for data analysis. The Mineral Resource estimation was completed by Mr Jeremy Witley, the Qualified Person for the Mineral Resource.

14.1 Mineral Resource Estimation Database

The Mineral Resource estimate was based on geochemical analyses and density measurements obtained from the cores of diamond drillholes, which were completed by KICO from March 2014–November 2015 and May–November 2017, with the cut-off date for data included in this estimate being 26 April 2018. As at the cut-off-date, there were no outstanding data of relevance to this estimate and the database was complete. In addition to the KICO drillholes, La Générale des Carrières et des Mines (Gécamines) drilled numerous diamond drillholes during the operational period of the mine, which were considered individually for inclusion into the estimate.

14.1.1 Gécamines Drillhole Database

The Gécamines database was compiled by capturing information from digital scans of hard copy geological logs. Information on the drillhole collar, downhole survey, lithology, sample assays and density were captured into Microsoft Excel spreadsheets and compiled into a Microsoft Access database by MSA. Databases had previously been compiled in a similar way by the Mineral Corporation (a South African consultancy) prior to MSA's involvement in the project. These databases were validated, revised, and additional data were added to encompass the full area of interest.

The scanned copies of the log sheets supplied to MSA consist of:

- Typed or handwritten geological logs with drillhole collar information on the sheet.
- Downhole survey reports – Survey readings were taken at approximately 50 m intervals, although not all of the holes have downhole survey data.
- Handwritten sample sheets with corresponding assay values.
- A Microsoft Excel sample sheet with corresponding assay data.

The degree of completeness of the hardcopy data was found to be variable and in many cases information such as assays or collar surveys was missing or incomplete. Assay data were generally contained in two hardcopy sheets, handwritten sample, and assay sheets, as well as computer print-out sheets. In many cases the computer print-out sheet represented composited data. The handwritten sample data were captured in favour of that in the computer print-out sheet.

The Gécamines collars were located in a local mine grid. In some cases, Gaussian coordinates were available and where not available the mine grid coordinates were converted to Gaussian coordinates and validated against the surveys of the underground workings.

The following data were captured in spreadsheets:

- Collar information,
- Downhole surveys – where there are no survey data for a drillhole, the collar survey inclination, and bearing were used as the downhole survey,
- Assays; grades of Cu, Pb, Zn, S, Fe, As, and density,
- Lithological log, and
- Mineralisation log.

Once the data were captured, the accuracy of the capturing was determined by checking 10% of the captured data against the hardcopy logs. The data were then checked for completeness to ensure that each drillhole record has corresponding records for collar, downhole survey, assay, lithology, and mineralisation. Missing aspects of the data were sought and captured if found. The maximum depth of each drillhole was compared across each of the tables to identify whether logs were complete. Any discrepancies were checked and rectified where appropriate.

Once the check for completeness was complete, the integrity of the data was checked:

- The drillhole name was compared to the level, section, and cubby number recorded in the collar table. Discrepancies were checked against hardcopy records and corrected where necessary.
- The dip of the drillhole is recorded in the drillhole name, this was compared to the dip from the survey sheets. Discrepancies were checked with the hardcopies and were corrected where necessary.
- Consistency in the drillhole name between tables was compared and where transcription errors or errors in the hard copy data were found, the drillhole names were modified appropriately.
- Duplicated logs were removed. Where duplicate data were found, the most complete sheet was used.
- Missing, duplicated, or overlapping intervals were identified by summing the length of intervals within a specific hole and comparing the sum to the depth in the collar table.
- The range of reported assays was checked to ensure that elements were consistently reported in percent or ppm as appropriate.

Once the data had passed the capturing validation tests it was imported into a Microsoft Access database for further checks. Of the drillholes imported, 33 did not have collar coordinates and the data from these holes were moved into a quarantined area of the database.

In total, 344 of the Gécamines drillholes were captured that passed the database checks.

14.1.2 KICO Drillhole Database

Ninety-seven diamond drillholes were completed by KICO from March 2014–November 2015 and a further 59 from May–November 2017. The data from these holes are stored in a Microsoft Access database that in the Qualified Person's opinion conforms to modern acceptable database management protocols. The information contained in the database is comprehensive and contains data tables for collar surveys, downhole surveys, lithology, structure, geotechnical measurements and observations, sample assays, and density.

Eight Gécamines drillholes were re-sampled by KICO. Infill sampling of these holes was also completed where Gécamines had not sampled the lower-grade intervals within the mineralised envelope. The original Gécamines data were replaced with the KICO re-sampled data for the purposes of the Mineral Resource estimate.

Eleven of the Gécamines holes were twin-drilled by KICO (Table 14.1). Where the holes were drilled within a few metres of one another, the Gécamines holes were discarded from the final database used for modelling. This was necessary as the KICO drillholes were completely sampled in the lower-grade mineralisation than the Gécamines holes and thus any short-range discontinuities in the lower-grade mineralisation due to different sampling protocols were avoided.

Table 14.1 Kipushi Twinned Holes

Gécamines Drillhole	Twinned with KICO Drillhole
1270/5/V+30/–45/SE	KPU046
1270/5/V+30/–65/SE	KPU064
1270/11/V+30/–65/SE	KPU062
1270/5/V+30/–55/SE	KPU059
1270/17/W/–35/SE	KPU070
1270/17/W/–76/SE	KPU069
1270/5/V+30/–75/SE	KPU057 and KPU051
1270/15/W/–20/SE	KPU068
1270/7/V+30/–75/SE	KPU051
1270/9/V+30/–63/SE	KPU071
1270/13/V+45/–30/SE	KPU065

The KICO sample assay database contains assay data for a number of elements as shown in Table 14.2.

Table 14.2 Assays in KICO Sample Database

Element	Units	Element Symbol	Lower Detection Limit
Gold	ppb	Au	1
Platinum	ppb	Pt	20/50
Palladium	ppb	Pd	20/50
Mercury	ppm	Hg	0.01/10
Silver	ppm	Ag	5 or 0.05
Arsenic	ppm	As	10
Cadmium	ppm	Cd	10
Cobalt	ppm	Co	10
Copper	ppm	Cu	50
Germanium	ppm	Ge	5
Lead	ppm	Pb	20
Zinc	ppm	Zn	50
Rhenium	ppm	Re	0.1
Sulfur	%	S	0.01
Nickel	ppm	Ni	20/50
Molybdenum	ppm	Mo	5
Uranium	ppm	U	0.5
Vanadium	ppm	V	20/50

Silver was first assayed using a single acid digest method, which has a lower detection limit of 5 ppm and 5 ppm precision. Where the initial silver assay returned a value of 50 ppm or less, the silver grade was determined again by Aqua Regia digest method, which is considered to be more accurate at lower levels. Hence, two records for silver were found in the database. In the final data used in the Mineral Resource estimate, the initial single acid digest values of 50 ppm or less were replaced by the Aqua Regia values.

Where the assay returned a value of less than the lower detection limit, the value was assigned a minus value in the database equivalent to the lower detection limit of that element multiplied by negative 1 (i.e. –0.1). For estimation purposes, all negative assays were re-assigned a zero value.

14.2 Exploratory Analysis of the Raw Data

14.2.1 Validation of the Data

A final validation exercise was completed by the Qualified Person for the Mineral Resource. The validation process consisted of:

- Examining the sample assay, collar survey, downhole survey, and geology data to ensure that the data was complete for all the drillholes.
- Examination of the assay and density data to ascertain whether they are within expected ranges.
- Examining the de-surveyed data in three dimensions to check for gross spatial errors and their position relative to mineralisation.
- Checks for 'from-to' errors, to ensure that the sample data do not overlap one another or that there are no unexplained gaps between samples.

The data validation exercise revealed the following:

- Below detection limit values were set to negative values in the database. All below detection limit assays were set to a value of zero for estimation purposes.
- There are intervals of Gécamines drill core that were not sampled or assayed. These intervals were set to zero grade on the assumption that there was no visible mineralisation worth sampling and thus the core interval is barren. The Gécamines cores were selectively sampled and samples were only taken when mineralisation was visibly determined to be above a threshold perceived to be economic at the time. For this reason, the assignment of zero grades to un-sampled intervals in the Gécamines database may be considered conservative, although this is the only reasonable option for the data.
- There are intervals of KICO drill core that were not sampled or assayed. These intervals were set to zero grade on the assumption that there was no visible mineralisation worth sampling and thus the core interval is barren. The KICO cores were mostly sampled throughout the length within the mineralised zones and the assignation of zero grades to un-sampled intervals will not result in any biases. For KPU075, a large part of the mineralised intersection was not sampled, it being used for metallurgical studies. For this hole the assays were set to null ('-') values where there are no sample assay data available within the mineralised zone (as observed by the mineralisation log).
- Seven of the drillholes from the recent KICO drilling programme were drilled for metallurgical purposes and were not sampled. For these holes the assay values were set to null ('-').
- Several holes were drilled to investigate elevated zinc–copper mineralisation outside of the main zones of mineralisation and were not included in the estimation data.
- The assay data available for the Gécamines holes varies in completeness. If the copper value is blank, the assays for each element were set to zero, including copper. Where a sample has copper and/or zinc values, but other assays are missing, these were set to null, and the copper and/or zinc values were retained.

- Several of the KICO specific gravity measurements are outside of expected limits. Ten measurements are less than 2.1 g/cm³ and were set to a null value ('-') by MSA. Twenty-two measurements are greater than 5.25 g/cm³ and were set to null values.
- There are no unresolved 'from-to' errors in the database.
- The assay values in the database are within expected limits for the Kipushi mineralisation. This is with the exception of a single silver value of 27,600 ppm that was discarded, the next highest value being 4,240 ppm.
- There are no assays at the upper detection limit that were not sent for over-limit assays.

Drillholes were discarded from the Gécamines database for a number of reasons:

- There are eight cases where an entire Gécamines drillhole had intersected the mineralised zone and no assays were captured. In each of these cases the drillhole was rejected from the estimation database.
- Four Gécamines drillholes appear to be incorrectly coordinated as they do not plot in the expected position relative to other holes and the Kipushi mineralised zones. These drillholes are 1132/18/V+6/–60/SE, which does not fit the mineralised zones.
- 1138/1/R+31/–70/SW which plots well within the Fault Zone footwall, 1138/1/R+31/–70/NW mineralised intercept plots well within the Série Récurrente footwall, and 1132/10/HZ/SE for which the geology is not consistent with the surrounding drillholes and does not fit the geological model. These four holes were not used in the modelling process.
- 1132/4/V+30/–55/SE has the same assay values in two adjacent intervals and so was discarded as it is likely this is erroneous. 1270/5/V+30/–85/SE has many of the same assay values in adjacent intervals and it appears the same long interval may have been divided into short intervals. This drillhole was discarded from the estimation database.
- Many of the Gécamines sample lengths appear excessive due to composited data (where sample lengths have been combined into longer intervals) being captured. Gécamines would take long samples (often 4 m or more) in homogenous mineralisation and so the data from each hole that contain excessive sample lengths (>4 m) were examined. The assays from these holes were flagged and not used for grade estimation if they appeared to be composited data. The composite sample hole data were used in the construction of the model to define the mineralisation extents but were not used in the estimation of the grade block model. In total, the assays from 131 Gécamines holes were not used for grade estimation.
- Fourteen Gécamines holes had been drilled along or close to the plane of the mineralisation either in dip or strike direction in the Série Récurrente zone. These holes were not used for grade estimation but were used for defining the extents of the mineralisation.
- The position of the mineralised zones in one hole (1132/4/U+30/–90) did not compare well with the surrounding KICO holes and was discarded from the estimation database.
- Eleven Gécamines holes had been twin-drilled and were removed in favour of the KICO drillholes.

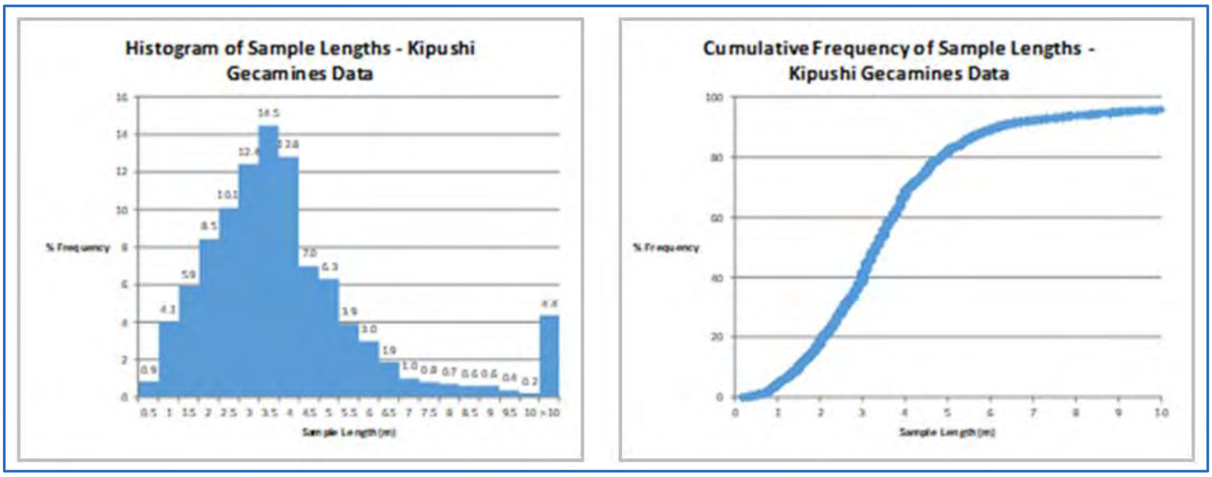
In total there are 134 KICO drillholes that have assays and intersected the targeted mineralised zones. 106 Gécamines drillholes were deemed acceptable for use in the grade interpolation process and an additional 144 Gécamines drillholes were included for the purpose of defining mineralisation limits.

The validated KICO and Gécamines data were combined for grade estimation. Consideration of the lack of certainty in the quality of the Gécamines data was made when classifying the Mineral Resource into the respective CIM categories of Measured, Indicated, and Inferred.

14.2.2 Statistics of the Sample Data

The Gécamines sample data were captured from scans of hard copy handwritten and digital logs. Gécamines tended to use a variety of sample lengths, considerably longer than what would normally be used in modern practice. In addition, as the database contains composite sample lengths, a number of extreme sample lengths were reported from the database with 4.4% of the sample lengths being greater than 10 m (Figure 14.1). The most frequent sample lengths are between 2–5 m and 82.5% of the sample records have a length of less than or equal to 5 m long. As mentioned in Section 14.2.1, Gécamines drillholes that contain well mineralised sample lengths that are excessive were flagged in the estimation database. These holes were used in the construction of the grade shell to define the mineralisation extents but were not used in the estimation of the grade block model.

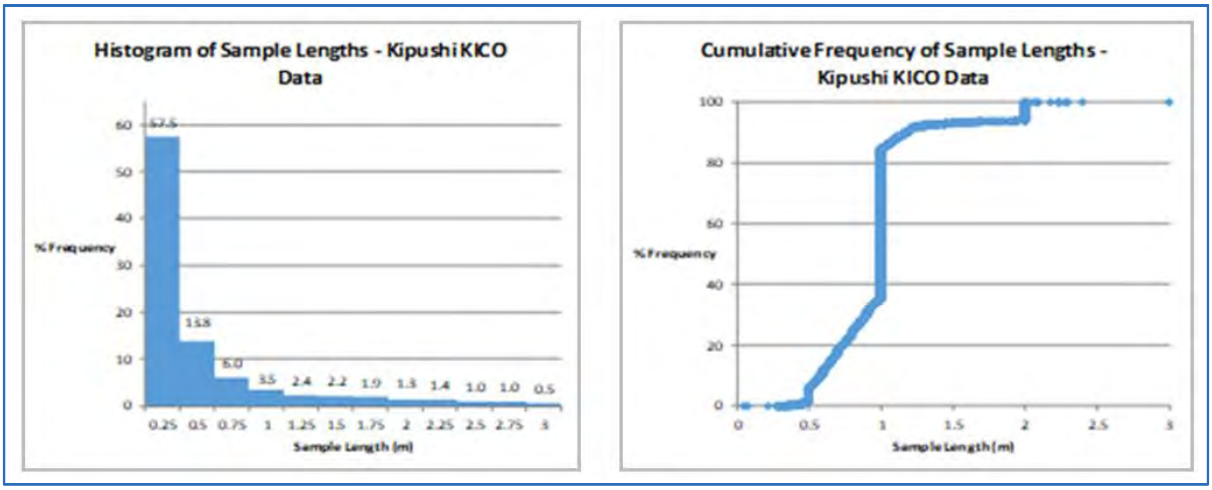
Figure 14.1 Histogram and Cumulative Frequency Plot of the Sample Length Data – Gécamines



MSA, 2018

The KICO sampling honoured the intensity of mineralisation and geological contacts. In homogenous zones nominal sample lengths of 1 m or 2 m were taken, with the longer samples tending to be taken from low-grade or waste zones (Figure 14.2).

Figure 14.2 Histogram and Cumulative Frequency Plot of the Sample Length Data – KICO



MSA, 2018

14.2.3 Statistics of the Assay Data

Platinum and palladium assays are of negligible grade, these assays being largely below the detection limit with rare instances of assays of 20 ppb, 40 ppb, or 60 ppb. The assays for gold are low and only 11 values are greater than 0.5 g/t and there are only 41 values above 0.2 g/t. Two samples returned assays of 2.72 g/t and 3.16 g/t Au respectively. Samples from drillholes completed in 2017 were not assayed for platinum, palladium, and gold.

Not all of the KICO samples were assayed for nickel, vanadium, or uranium. The earlier drillholes completed by KICO were assayed for nickel and vanadium but, due to the low values experienced, they were discontinued from KPU030 onwards. KPU001 and KPU002 were not assayed for uranium.

The highest nickel assay is 200 ppm with the majority of the values being below the lower detection limit. Most of the vanadium values are below or slightly above the lower detection limit with the maximum assay being 640 ppm.

As the assays for Pt, Pd, Au, Ni, and V are of negligible grade, these elements were not considered further in the Mineral Resource estimate.

The KICO samples were also assayed for mercury, uranium, molybdenum, and rhenium. Some of the samples have significant grades for these elements, but overall, they are low. Mercury assays are less than 200 ppm. Sixty-three percent of the molybdenum assays are below the lower detection limit (5 ppm), and only 16 values are above 1,000 ppm. Seventy-one percent of the rhenium assays are below the lower detection limit (0.10 ppm) and only eight values are greater than 50 ppm. Uranium values are generally low with approximately 98% of the values being below 10 ppm, 29 values being above 50 ppm and the maximum assay being 513 ppm. Given the low numbers of significant assays for Hg, Mo, and Re these elements were not considered further in the Mineral Resource estimate, as the value that they could contribute to the project is insignificant. Uranium may be considered a nuisance or deleterious element in situations where it exists in amounts too low to derive economic value. It is uncertain whether the amount of uranium at Kipushi will be of any impact to the project given the generally low values.

Copper, lead, zinc, sulfur, arsenic, silver, germanium, cobalt, cadmium, iron, and density were considered of importance to the Kipushi Project. As a result, these were examined in greater detail and estimated into the Mineral Resource block model.

14.2.3.1 Univariate Analysis

A summary of the sample assay statistics of the un-composited data at Kipushi is shown in Table 14.3 for the Gécamines data and Table 14.4 for the KICO data.

Table 14.3 Summary of the Raw Validated Sample Data*1 for the Gécamines Drillholes

Variable	Number of Assays	Mean Value	Minimum Value	Maximum Value
Cu %	2,182	2.41	0.005	60.80
Pb %	2,178	0.52	0.005	16.40
Zn %	2,182	9.91	0.005	63.15
S %	1,926	12.84	0.03	43.65
As %	1,823	0.17	0.005	7.46
Ag g/t	No Data	–	–	–
Ge g/t	No Data	–	–	–
Co ppm	No data	–	–	–
Cd ppm	No Data	–	–	–
Fe %	1,920	8.29	0.78	39.01

*1 Where re-sampled Gécamines assays have been replaced with KICO assays.

Table 14.4 Summary of the Raw Validated Sample Data for the KICO Drillholes

Variable	Number of Assays	Mean Value	Minimum Value	Maximum Value
Cu %	11,064	1.11	0.00	40.40
Pb %	11,064	0.17	0.00	17.90
Zn %	11,064	12.11	0.00	65.20
S %	11,064	12.10	0.00	51.70
As %	11,064	0.18	0.00	14.70
Ag g/t	11,063	13.00	0.00	4,240
Ge g/t	11,064	33.70	0.00	19,600
Co ppm	11,064	38.40	0.00	25,300
Cd ppm	11,064	688	0.00	14,500
Fe %	10,886	6.49	0.19	51.90
Density g/cm^3	10,402	3.28	2.03	5.22

The Gécamines database does not contain values for silver, germanium, copper, or cadmium as well as some of the copper, lead, zinc, sulfur, iron, and arsenic values. The mean assay values for the KICO copper and lead data are less than those of the Gécamines data as the KICO cores were completely sampled in the potentially mineralised zones, unlike the Gécamines sampling that was selective aimed at higher grade copper or zinc mineralisation.

Several zones of mineralisation have been identified by Gécamines and KICO. The zones of mineralisation are either copper dominant or zinc dominant with varying amounts of other elements. The grade distributions are characterised by large amounts of low-grade data (below approximately 0.2% for copper and 5% for zinc), medium grade data and high-grade (above approximately 20% for copper and 20% for zinc) data.

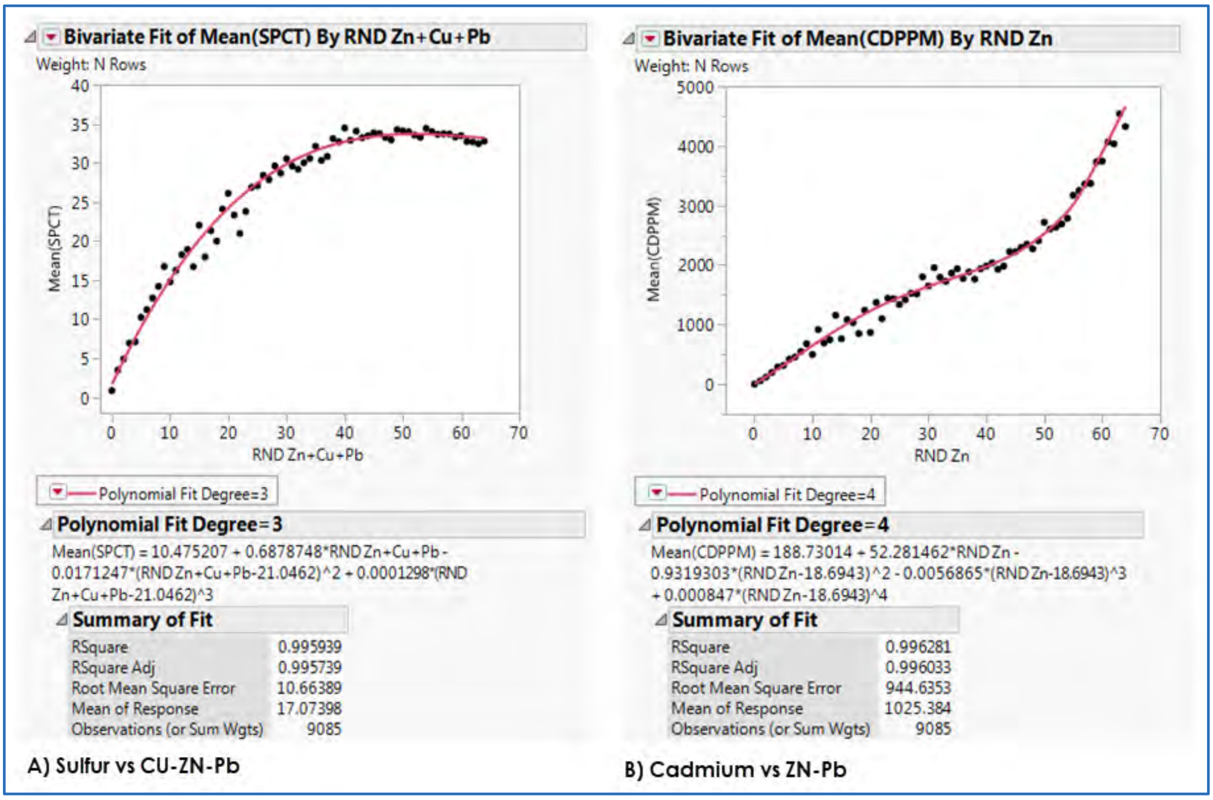
Bivariate Analysis

Scatterplots were made that compare the grades of individual elements against one another. The scatterplots for the total data show various relationships that indicate mixed mineralisation domains. Several mineralisation styles at Kipushi exist, the zinc-rich zones resulting in different bivariate relationships than the copper-rich zones. No clear relationships were found between copper, lead, zinc, and cobalt. Mixed linear relationships are evident between copper and sulfur, zinc and sulfur, copper and density, and zinc and density. The zones tending to be either copper or zinc rich. The strongest relationships are observed between lead and silver, zinc and germanium, and sulfur and density. A very strong relationship was observed between zinc and cadmium.

Regression for Un-assayed Elements

There is a strong relationship between copper-lead-zinc and sulfur, and between zinc and cadmium. Sulfur assays are not always present in the Gécamines samples and there are no cadmium assays at all in the Gécamines dataset. For these elements a regression formula was applied to the missing data to ensure that the relationships between them are locally preserved in the estimate (Figure 14.3). A third order polynomial line was fitted to the sulfur vs copper-lead-zinc regression and a fourth order polynomial line was fitted to the cadmium vs zinc regression. Missing values for elements that do not have a strong relationship between one another were left as missing (null) values in the estimation data.

Figure 14.3 Sulfur and Cadmium Regressions



MSA, 2018

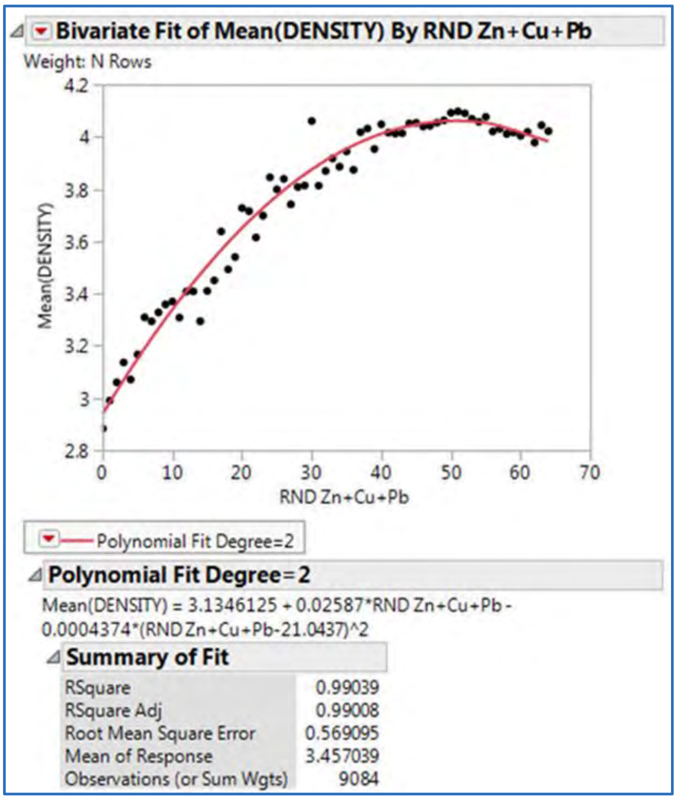
Density Determination

Density was measured by KICO on whole lengths of half core samples, using Archimedes principal of weight in air versus weight in water. Not all the KICO samples were measured for density. Many of the Gécamines density values were derived from a calculation or considered unreliable and so the Gécamines density values were discarded. A regression was formulated from the KICO measurements, in order to estimate the density of each sample based on its grade. This formula was applied to all the Gécamines samples and to the KICO samples that did not have density measurements performed on them. It was found that a summation of copper, zinc, and lead grade versus density produced a reasonable regression for the multi-element mineralisation at Kipushi, however, the mineralisation at Kipushi is complex and it was difficult to produce a perfect fit for all grade ranges.

A second order polynomial curve was fitted to the data as shown in Figure 14.4, 52% Cu–Zn-Pb a slight decrease in density was observed with increasing grade.

It should be noted that use of regression formulae is not ideal and local biases will occur, however, it is expected that on average the density for each zone will be accurate.

Figure 14.4 Density Regression



MSA, 2018

14.2.4 Summary of the Exploratory Analysis of the Raw Dataset

- KICO assays below the detection limit were assigned zero values, they exist as negative values in the original database. The below detection values for the Gécamines data were retained at the very low, but positive, values existing in the data.
- Intervals of KICO core that were not sampled or assayed were assigned zero values for each of the elements of interest. This is with the exception of KPU075, for which a large part of the mineralised intersection was not sampled, it being used for metallurgical studies. For this hole the assays were set to null values where there are no sample assay data available within the mineralised zone as defined by the mineralisation log. Seven holes were drilled specifically for metallurgical test-work. These were also assigned null values.
- The assay data available for the Gécamines holes varies in completeness. If the copper value is blank, the assays for each element were set to zero including copper. Where a sample has copper and/or zinc values, but other assays are missing, the other values were set to null and the copper and/or zinc values were retained. This is based on the assumption that the missing values were not assayed and assigning zero value to them would be incorrect.
- Drillholes were discarded from the Gécamines database for a number of reasons, such as no assays captured, incorrect coordinates, excessive samples lengths due to composite data being captured, and inappropriate drilling directions. Gécamines holes that had been twin-drilled by KICO were also removed from the estimation data set.
- In total, there are 134 KICO drillholes that have sampling data. 106 Gécamines drillholes were deemed acceptable for use in the grade interpolation process and an additional 144 Gécamines drillholes were included for the purpose of defining mineralisation limits.
- The quality of the Gécamines data is less certain than for the KICO data. Consideration of this was made when classifying the Mineral Resource into the respective CIM categories of Measured, Indicated, and Inferred.
- Copper, lead, zinc, sulfur, arsenic, silver, germanium, cobalt, cadmium, and density are considered of importance to the Kipushi Project. A number of other elements were assayed by KICO; however, their concentrations are not significant. Uranium may be considered a nuisance or deleterious element in situations where it exists in amounts too low to derive economic value. It is uncertain whether the amount of uranium at Kipushi will impact the project at the low-grades in which it occurs.
- Missing values for sulfur and cadmium were assigned based on regression analysis in order to maintain the strong relationships observed between them and other groups of metals.
- Density measurements taken by KICO on core samples were used to generate a regression with copper, lead, and zinc and the regressed values were assigned to those KICO samples that did not have density measurements performed on them and all of the Gécamines samples.
- Several zones of mineralisation have been identified, either copper-rich or zinc-rich. These are spatially separate and need to be considered as separate domains in estimation.

14.3 Geological Modelling

14.3.1 Mineralised Zones

The mineralisation at Kipushi comprises sulfide replacement bodies within the Kakontwe Sub-Group dolomites and Série Récurrente Sub-Group dolomitic shales of the Nguba Group.

Two zones of zinc-rich mineralisation occur, the Big Zinc and the Southern Zinc, which lie adjacent to the copper-rich Fault Zone mineralisation. In places, the Big Zinc mineralisation is juxtaposed against the Fault Zone, although in many areas zones devoid of significant mineralisation occur between them. A zone of high-grade copper, silver and germanium occurs within the Big Zinc. The Southern Zinc zone is an elongate lense of sphalerite rich mineralisation parallel and juxtaposed against the Fault Zone mineralisation. The Southern Zinc becomes copper-rich and zinc-poor towards the south.

The Fault Zone strikes north–north-east–south–south-west and dips at approximately 70° to the west, with the zinc mineralisation forming irregular steeply dipping bodies in the immediate footwall to the Fault Zone. A low-grade zone occurs in the Fault Zone in the area between the Big Zinc and the Southern Zinc. A zone of high-grade copper-rich mineralisation occurs immediately adjacent to the Série Récurrente and strikes from east–west, is sub-vertical and plunges steeply to the west. This zone transgresses into the Série Récurrente in places. Where the Fault Zone and Série Récurrente meet, mineralisation tends to be enhanced in a sub-zone known as the Copper Nord Riche. A sub-vertical copper–zinc–germanium rich sulfide zone occurs as a splay from the Fault Zone at depth towards the south-west.

Significant concentrations of lead, silver, cobalt, and germanium occur in variable amounts in all zones.

Although there are distinct lithological and structural controls to the mineralisation, a characteristic of the replacement nature of the mineralisation is that it cuts across the layering in places and is not stratabound. For this reason, the mineralisation was modelled on the basis of grade thresholds while taking cognisance of the controlling lithological and structural trends.

In total, ten zones were modelled as separate wireframes:

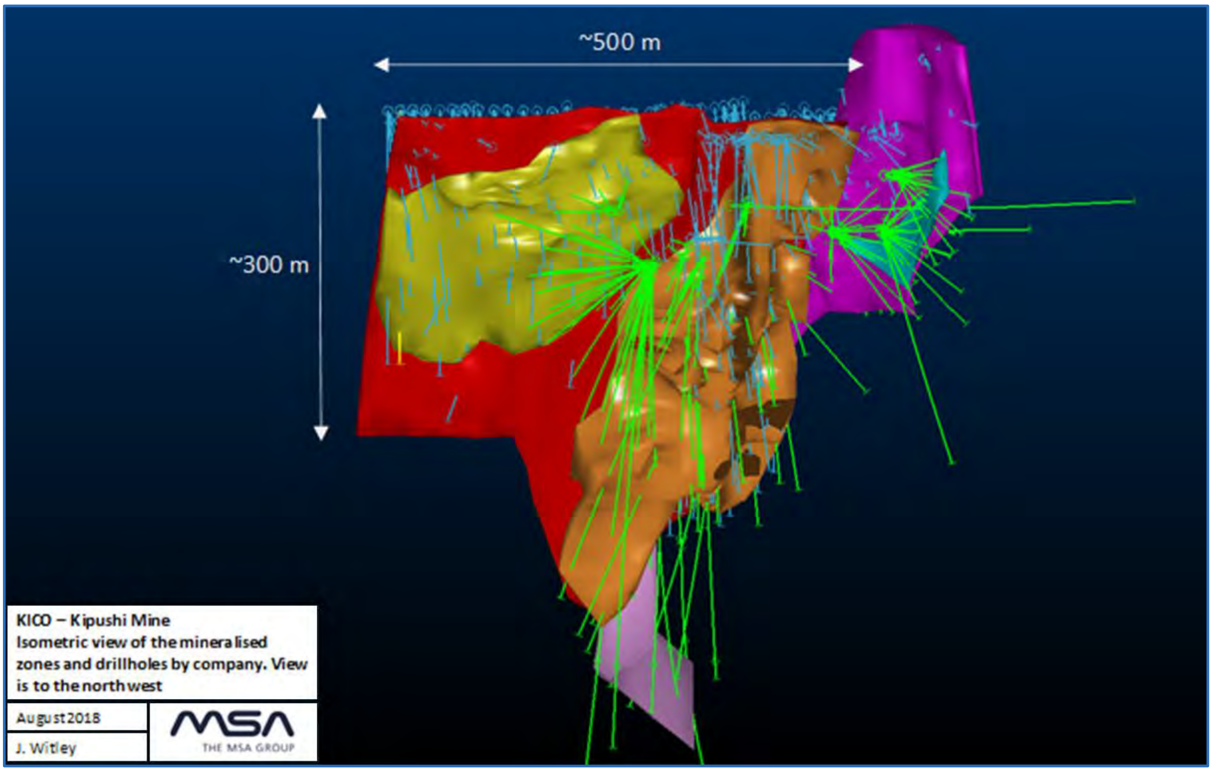
- Fault Zone – Zone 1. A low-grade zone (Zone 9) was defined within the Fault Zone.
- Big Zinc – Zone 2.
- Southern Zinc – Zone 3. A low zinc, moderate copper zone (Zone 10) was defined towards the south.
- Série Récurrente – Zone 4.
- Massive sulfide lens adjacent to Série Récurrente – Zone 5. A massive sulfide lens occurs within it (Zone 8).
- High-grade copper zone within the Big Zinc – Zone 6.
- Fault Zone Splay – the high zinc-copper-germanium splay from the Fault Zone – Zone 7.

Mineralised zones were identified using a threshold value of 5% for zinc and 1.0% for copper. Strings were constructed along sections perpendicular to the dip of the mineralisation by snapping to the drillhole intercepts. The sections were examined along strike to ensure that the thickness trends of the mineralisation were continued from one section to the next. The interpreted strings were then linked to form wireframe solids.

All available validated data were used for the construction of the mineralised models. The Gécamines drillholes that were rejected from the grade estimation due to excessive sample lengths were also used.

The resulting wireframe shells show local irregularities although clear trends are evident, particularly for the Big Zinc that plunges steeply to the south-west. An isometric view of the wireframe models is shown in Figure 14.5.

Figure 14.5 Isometric View of Kipushi Wireframes and Drillholes (view is approximately to the north-west)



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Red Wireframe = Fault Zone (Zone 1).
Orange Wireframe = Big Zinc (Zone 2).
Yellow Wireframe = Southern Zinc (Zone 3).
Magenta Wireframe = Série Récurrente (Zone 4).
Cyan Wireframe = High-grade copper zone adjacent to Série Récurrente (Zone 5).
Pink Wireframe = Fault Zone (Zone 7).
Blue traces =Gécamines drillholes.
Green traces = KICO drillholes.

14.4 Statistical Analysis of the Composite Data

The drillhole sample data that were considered suitable for estimation purposes were selected by zone using the modelled wireframes and then composited to 2 m lengths using density-length weighting. The composites were de-clustered to a cell size of 20 m X, 20 m Y, and 20 m Z by weighting by the number of data in each cell and summary statistics were compiled for each mineralised zone (Table 14.5).

The summary statistics were interrogated, paying particular attention to the variability (as exhibited by the coefficient of variation (CV)) and the skewness, as high skewness tends to be an indication of a number of particularly high-grade values within a generally lower-grade distribution.

Table 14.5 Summary Statistics (de-clustered) of the Estimation 2 m Composite Data for Grades and SG

Variable	Number of composites	Minimum	Maximum	Mean	CV	Skewness
Zone 1 – Fault Zone						
Cu %	601	0.00	42.25	3.53	1.23	3.1
Pb %	601	0.00	3.72	0.13	3.25	5.9
Zn %	601	0.00	43.83	3.77	1.80	3.2
S %	601	0.00	50.01	14.66	0.76	0.7
As %	482	0.00	9.33	0.45	1.89	4.7
Ag g/t	280	0.00	165.8	20.6	1.33	3.0
Ge g/t	280	0.00	433.7	35.5	1.62	4.5
Co ppm	280	0.00	13,121	243	5.02	7.6
Cd ppm	601	0.00	6,776	273	1.97	5.1
Density	617	2.69	4.67	3.24	0.10	1.2
Iron %	503	0.00	44.45	14.1	0.75	0.76
Zone 2 – Big Zinc						
Cu %	2,913	0.00	60.80	0.97	3.23	8.2
Pb %	2,913	0.00	16.77	0.80	2.75	3.8
Zn %	2,913	0.00	63.43	31.32	0.63	–0.3
S %	2,913	0.00	45.72	25.03	0.50	–0.9
As %	2,878	0.00	5.54	0.16	2.09	8.9
Ag g/t	2,105	0.00	196.1	14.7	1.33	3.3
Ge g/t	2,105	0.00	655.8	52.5	1.04	4.3
Co ppm	2,105	0.00	5,483	23	8.4	23.1
Cd ppm	2,913	0.00	5,557	1,543	0.72	0.4

Variable	Number of composites	Minimum	Maximum	Mean	CV	Skewness
Density	3,148	2.19	4.82	3.73	0.12	–0.7
Iron %	2,850	0.00	40.66	9.33	0.70	1.0
Zone 3 – Southern Zinc						
Cu %	217	0.00	28.71	2.95	1.34	2.8
Pb %	217	0.00	9.72	1.47	1.35	1.8
Zn %	217	0.00	61.7	24.31	0.67	0.05
S %	217	0.00	40.50	24.92	0.44	–1.0
As %	128	0.00	2.51	0.37	1.12	3.1
Ag g/t	103	1.60	3,106	131.1	2.96	5.5
Ge g/t	103	0.00	12,704.9	442.1	3.71	6.0
Co ppm	103	0.00	110	7	1.91	4.8
Cd ppm	217	0.00	14,273	2,880	1.20	1.7
Density	230	2.84	4.59	3.73	0.11	–0.5
Iron %	127	0.00	36.80	12.05	0.67	0.7
Zone 4 – Série Récurrente						
Cu %	1,453	0.00	26.75	1.96	1.39	4.4
Pb %	1,453	0.00	1.94	0.03	5.18	9.7
Zn %	1,453	0.00	32.14	0.65	3.96	7.3
S %	1,453	0.00	35.61	2.82	1.66	4.0
As %	1,419	0.00	1.70	0.06	2.30	6.7
Ag g/t	583	0.00	96.8	7.9	1.05	3.4
Ge g/t	583	0.00	9.1	0.6	2.62	2.9
Co ppm	583	0.00	1,366	36	2.52	8.0
Cd ppm	1,453	0.00	1,714	41	3.75	6.7
Density	1,453	2.39	4.05	3.02	0.06	2.8
Iron %	1,453	0.00	32.89	3.64	0.96	4.3
Zone 5 – Série Récurrente Footwall						
Cu %	78	0.18	5.40	1.78	0.63	1.6
Pb %	78	0.00	11.26	0.12	8.10	10.5
Zn %	78	0.00	54.29	5.02	2.67	2.7
S %	78	0.47	29.33	4.76	1.65	2.4
As %	78	0.00	0.69	0.08	1.62	3.7
Ag g/t	78	0.00	60.4	11.5	1.08	2.3

Variable	Number of composites	Minimum	Maximum	Mean	CV	Skewness
Ge g/t	78	0.00	28.5	3.16	2.10	2.3
Co ppm	78	0.00	1,229	30	3.85	9.6
Cd ppm	78	0.00	3,722	311	2.77	2.9
Density	82	2.82	4.07	3.08	0.10	2.2
Iron %	78	0.76	16.05	2.56	0.98	4.6
Zone 6 – High-grade copper zone within Big Zinc						
Cu %	115	0.88	32.53	6.64	0.85	1.7
Pb %	115	0.00	13.10	1.03	2.26	3.7
Zn %	115	0.01	54.90	26.86	0.67	–0.2
S %	115	1.20	42.05	26.24	0.45	–1.0
As %	115	0.01	0.86	0.19	0.89	2.1
Ag g/t	89	7.60	2,036.7	175.4	2.0	3.2
Ge g/t	89	5.70	410.0	75.0	0.92	3.2
Co ppm	89	0.00	4,577	182	2.76	6.4
Cd ppm	115	0.00	3,724	1,627	0.70	0.0
Density	138	2.67	4.79	3.81	0.12	–1.0
Iron %	115	1.19	31.45	11.20	0.60	0.8
Zone 7 – Fault Zone Splay						
Cu %	94	0.00	20.16	2.88	1.43	2.1
Pb %	94	0.00	0.09	0.01	1.88	3.1
Zn %	94	0.01	64.27	27.86	0.98	0.1
S %	94	0.48	38.83	26.49	0.39	–1.4
As %	94	0.00	12.43	2.26	1.45	1.5
Ag g/t	94	0.10	82.3	14.8	1.00	2.1
Ge g/t	94	0.00	599.8	144.3	1.13	0.8
Co ppm	94	0.00	2,210	98	2.36	7.1
Cd ppm	94	0.00	6,189	2,007	1.03	0.4
Density	94	2.86	4.63	3.76	0.12	–0.7
Iron %	94	0.55	40.35	12.74	0.98	0.8
Zone 8 – Massive Sulfide in Série Récurrente Footwall						
Cu %	70	0.63	35.45	14.56	0.62	0.5
Pb %	70	0.00	5.42	0.21	3.70	4.8
Zn %	70	0.00	52.96	8.28	1.78	1.8

Variable	Number of composites	Minimum	Maximum	Mean	CV	Skewness
S %	70	0.59	31.67	19.12	0.47	–0.4
As %	70	0.00	5.35	0.58	1.95	3.3
Ag g/t	70	0.00	1,205.8	135.5	1.69	3.3
Ge g/t	70	0.00	67.5	16.0	1.04	1.3
Co ppm	70	0.00	5,182	259	3.14	5.5
Cd ppm	70	0.00	4,319	542	1.85	2.0
Density	73	2.87	4.20	3.53	0.10	–0.2
Iron %	70	0.96	27.26	13.49	0.55	0.2
Zone 9 – Low-grade zone in Fault Zone						
Cu %	37	0.00	4.87	0.56	1.37	4.1
Pb %	37	0.00	3.04	0.21	2.85	4.0
Zn %	37	0.00	42.79	7.12	1.32	2.7
S %	37	0.00	39.26	13.37	0.82	0.9
As %	34	0.00	2.05	0.26	1.95	3.1
Ag g/t	11	2.70	17.3	6.7	0.63	1.3
Ge g/t	11	5.00	38.9	15.5	0.63	1.1
Co ppm	11	0.00	24	12	0.72	-0.2
Cd ppm	37	0.00	2,536	491	1.11	2.1
Density	37	2.83	4.14	3.23	0.10	1.2
Iron %	34	0.00	34.84	10.69	0.81	1.2
Zone 10 – Southern portion of Southern Zinc						
Cu %	55	0.00	6.78	1.89	0.75	1.7
Pb %	55	0.00	0.51	0.05	1.88	3.3
Zn %	55	0.00	14.45	1.57	1.58	3.2
S %	55	0.00	40.46	8.51	1.24	2.4
As %	3	0.00	0.00	0.00	–	–
Ag g/t	0	–	–	–	–	–
Ge g/t	0	–	–	–	–	–
Co ppm	0	–	–	–	–	–
Cd ppm	55	0.00	912	101	155	3.2
Density	55	2.94	3.49	3.08	0.04	1.5
Iron %	19	0.00	34.35	9.16	1.45	1.4

For each element in most domains there are a significant number of composites with zero-grade. These largely represent un-sampled intervals within the mineralisation wireframes, many of which are derived from Gécamines sample data for which sampling was selective. There are no silver, germanium, and cobalt data available for the southern portion of the Southern Zinc (Zone 10), this zone being informed only by Gécamines data.

The copper distributions are generally characterised by moderate CV and are slightly positively skewed. Copper in Zone 2 (the Big Zinc) has a high CV and is strongly positively skewed. The zinc distributions in the zinc-rich zones show low to moderate CVs and have near symmetrical distributions and low kurtosis (i.e. have a flat shape). Zinc distributions in the other zones are variable, with high CVs in the copper-rich zones, but low to moderate in the high-grade more massive copper-rich sulfide zones (Zone 5 and 6). Cadmium exhibits similar distributions as zinc.

The CVs for lead are moderate to high and distributions are strongly positively skewed, they generally consisting of a small number of high-grade values in a dominantly low-grade population.

Sulfur generally has low to moderate CVs, is negatively skewed in the massive sulfide zones (Zones 2, 3, 5, and 6) and is positively skewed in the relatively lower sulfur grade copper-dominant zones (Zones 1 and 4).

Arsenic is strongly positively skewed except in Zone 6 and Zone 3, where CVs are low to moderate and the skewness is moderate. The strong positive skewness is caused by a small number of particularly high values in the distributions. Mean arsenic grades vary between 0.06–0.58% except for the Fault Zone Splay (Zone 7), which is high in arsenic and the mean arsenic grade is 2.26%.

The silver distributions have moderate CVs and strong skewness as a result of a small number of extremely high values. Mean silver grades are particularly high in the massive chalcopyrite-rich zones (Zones 6 and 8).

Germanium CVs are low, and distributions are moderately positively skewed except for Zone 4 and 5 that are generally of low germanium grade with a few values significantly higher than the mean value. Mean germanium values are high in the Big Zinc and the massive chalcopyrite and bornite rich zone (Zone 6) within the Big Zinc. Particularly high germanium values occur in the Fault Zone Splay (Zone 7) and the Southern Zinc contains a number of very high germanium grade samples (>1,000 g/t).

Cobalt distributions are positively skewed with high CVs caused by a small number of high values within a generally low-grade population.

Density distributions are generally slightly negatively skewed in the massive sulfide zones and slightly positively skewed in the lower-grade copper-rich zones. CVs are low and the skewness is not severe.

The moderate CVs indicate that a linear method, such as ordinary kriging, is appropriate to estimate the grades. The zones with high CV's and that are strongly positively skewed are a result of a small number of high-grade values that can be considered outliers and measures that control their impact are required.

14.4.1 Outlier Control

The log probability plots and histograms of the composite data were examined for outlier values that have a low probability of re-occurrence, particularly where a small proportion of high-grade data makes up a disproportional amount of the domain mean, populations with high CVs and histograms with long tails. The outlier values identified were capped to a threshold as shown in Table 14.6. The threshold was set at the next highest value below the lowest identified outlier value. Decisions on the capping threshold were guided by breaks in the cumulative log probability plots and the location of the high-grade samples with respect to other high-grade samples.

The capping reduced the extreme CVs but several remained high (>2) in Zone 2 and Zone 4 where the distributions exhibit high skewness.

The lead, arsenic, silver, germanium, and cobalt distributions are characterised by small numbers of high-grade values within dominantly low-grade populations. The high-grades tend to occur in clusters. In order to retain the high-grade values locally, without smearing of the values throughout their respective estimation domains, a restricted omnidirectional search of 7 m was applied on the data during interpolation without capping applied. This allows the high-grades to influence only the block in which they occur and the immediately surrounding blocks. The estimates using the uncapped data replaced the estimates using the capped data. The parameters used for the restricted search are described in Section 14.7.2.

Table 14.6 Values Capped and their Impact on Sample Mean and CV

Attribute	Before Capping			After Capping			
	Number of Composites	Mean	CV	Cap Value	Number of Composites	Mean	CV
Zone 1 – Fault Zone							
Cu %	601	3.53	1.23	24.3	2	3.51	1.18
Pb g/t	601	0.13	3.25	0.56	31	0.08	1.92
Zn %	601	3.77	1.80	28.0	14	3.59	1.66
As %	482	0.45	1.89	3.05	10	0.41	1.54
Ge g/t	280	35.5	1.62	116	5	29.7	1.06
Co ppm	280	243	5.02	323	21	62	1.40
Cd ppm	601	273	2.0	1938	11	255	1.64
Zone 2 – Big Zinc							
Cu %	2,913	0.97	3.34	9.93	56	0.78	2.39
Pb %	2,913	0.80	2.75	9.97	39	0.76	2.60
Ag g/t	2,878	0.16	2.09	1.58	29	0.15	1.44
Ge g/t	2,105	52.5	1.04	280	12	51.3	0.90
Co ppm	2,105	23	8.41	104	59	11	2.02

Attribute	Before Capping			After Capping			
	Number of Composites	Mean	CV	Cap Value	Number of Composites	Mean	CV
Zone 3 – Southern Zinc							
Cu %	118	1.85	1.12	8.3	1	1.82	1.05
As %	217	2.95	1.34	16.6	5	2.87	1.25
Ag g/t	128	0.37	1.12	0.95	5	0.33	0.80
Ge g/t	103	131.1	2.96	171	16	56.0	0.96
Co ppm	103	442.1	3.71	813	14	160.3	1.49
Zone 4 – Série Récurrente							
Cu %	1,453	1.96	1.39	17.2	8	1.93	1.31
Pb g/t	1,453	0.03	5.18	0.19	35	0.01	2.40
Zn %	1,453	0.65	3.96	3.85	67	0.37	2.41
As %	1,419	0.06	2.30	0.34	30	0.05	1.48
Ag g/t	583	7.9	1.05	40	6	7.8	0.95
Ge g/t	583	0.6	2.62	5.5	10	0.6	2.50
Co ppm	583	36	2.52	181	15	28	1.37
Cd ppm	1,453	41	3.75	240	67	24	2.34
Zone 5 – Série Récurrente Footwall							
Pb g/t	78	0.12	8.10	0.011	5	0.002	2.02
As %	78	0.08	1.62	0.26	2	0.06	1.10
Ag g/t	78	11.5	1.08	40	2	10.9	0.95
Co ppm	78	30	3.85	134	3	20	1.60
Fe %	78	2.56	0.98	5.05	1	2.25	0.48
Zone 6 – High-grade Copper Zone within Big Zinc							
Pb %	115	1.03	2.26	4.15	8	0.75	1.66
Ag g/t	89	175.4	2.00	281	12	86.4	1.04
Ge g/t	89	75.0	0.92	177	2	68.0	0.62
Co ppm	89	182	2.76	672	7	125	1.51
Zone 7 – Fault Zone Splay							
Pb %	94	0.008	1.88	0.05	1	0.008	1.67
Co ppm	94	98	2.36	394	3	77	1.32

Attribute	Before Capping			After Capping			
	Number of Composites	Mean	CV	Cap Value	Number of Composites	Mean	CV
Zone 8 – Massive Sulfide in Série Récurrente Footwall							
Pb %	70	0.21	3.70	0.042	7	0.01	1.57
As %	70	0.58	1.95	2.44	2	0.46	1.49
Ag g/t	70	135.5	1.69	698	1	121.9	1.42
Co ppm	70	259	3.14	552	3	130	1.20
Zone 9 – Low-grade Zone in Fault Zone							
Cu %	37	0.56	1.37	1.81	2	0.50	0.95
Pb %	37	0.21	2.85	0.7	5	0.11	1.91
Zn %	37	7.12	1.32	17.5	2	5.95	0.95
As %	34	0.26	1.95	0.38	3	0.14	0.92
Zone 10 – Southern Portion of Southern Zinc							
Pb %	55	0.05	1.88	0.07	9	0.03	0.97
Zn %	55	1.57	1.57	6.2	2	1.39	1.26
S %	55	8.51	1.24	19.7	5	6.67	0.81
Cd ppm	55	101	155	399	4	89	1.23

14.5 Geostatistical Analysis

14.5.1 Variograms

The 2 m composite data were examined using variograms that were calculated and modelled using Snowden Supervisor software. Most attributes were transformed to normal scores distributions and the spherical variogram models were back-transformed to normal statistical space for use in the grade interpolation process.

Variograms were calculated on the 2 m composite data and modelled within the plane of mineralisation with the minor direction being across strike. Rotations were aligned within each zone for all the attributes estimated. Normalised variograms were calculated so that the sum of the variance (total sill value) is equal to one.

Variograms were modelled with either one, two or three spherical structures. The nugget effect was estimated by extrapolation of the first two experimental variogram points (calculated at the same lag as the composite length) to the Y axis.

For the Fault Zone and Southern Zinc, a plunge of 70° to the south-west within the plane of mineralisation was modelled. A plunge of 50° to the west was modelled for the Série Récurrente and a vertical plunge was modelled for the Big Zinc grade continuity. Although the limits of the Big Zinc plunge steeply to the south-west, this trend was not evident in the grade continuity analysis. The plunge directions of the major zones were maintained for the minor zones, with the exception of the Série Récurrente Footwall that has a plunge of 40° to the west. The directions of continuity were kept the same for all attributes within their respective zones.

There were insufficient data to calculate robust variograms for the Fault zone (Zone 7) and a variogram with a nugget effect of 0.03, a sill of 0.97 and a range of 40 m strike, 40 m dip, and 10 m across strike was applied. The same variogram was used for the low-grade copper zone in the Fault Zone (Zone 9) as for the main mineralised portion (zone 1). The Série Récurrente Footwall zones (Zone 5 and Zone 8) were combined for variography purposes.

For the Big Zinc and Série Récurrente the variogram models are robust and there being a number of experimental points at the chosen lag informing the model within the range of the variogram. Fault Zone variograms tend to be more erratic with less well-developed structure. The variograms for the smaller zones (Zone 3 and 5 to 10) are less robust as there being fewer composites in these zones.

For all zones and attributes, the variogram ranges are in excess of the general drillhole spacing.

The variogram model parameters are shown in Table 14.7, after the variance has been back transformed from normal scores and examples of normal scores variograms are shown in Figure 14.6 for Zone 2.

Figure 14.6 Zone 2 (Big Zinc) Zinc Variograms

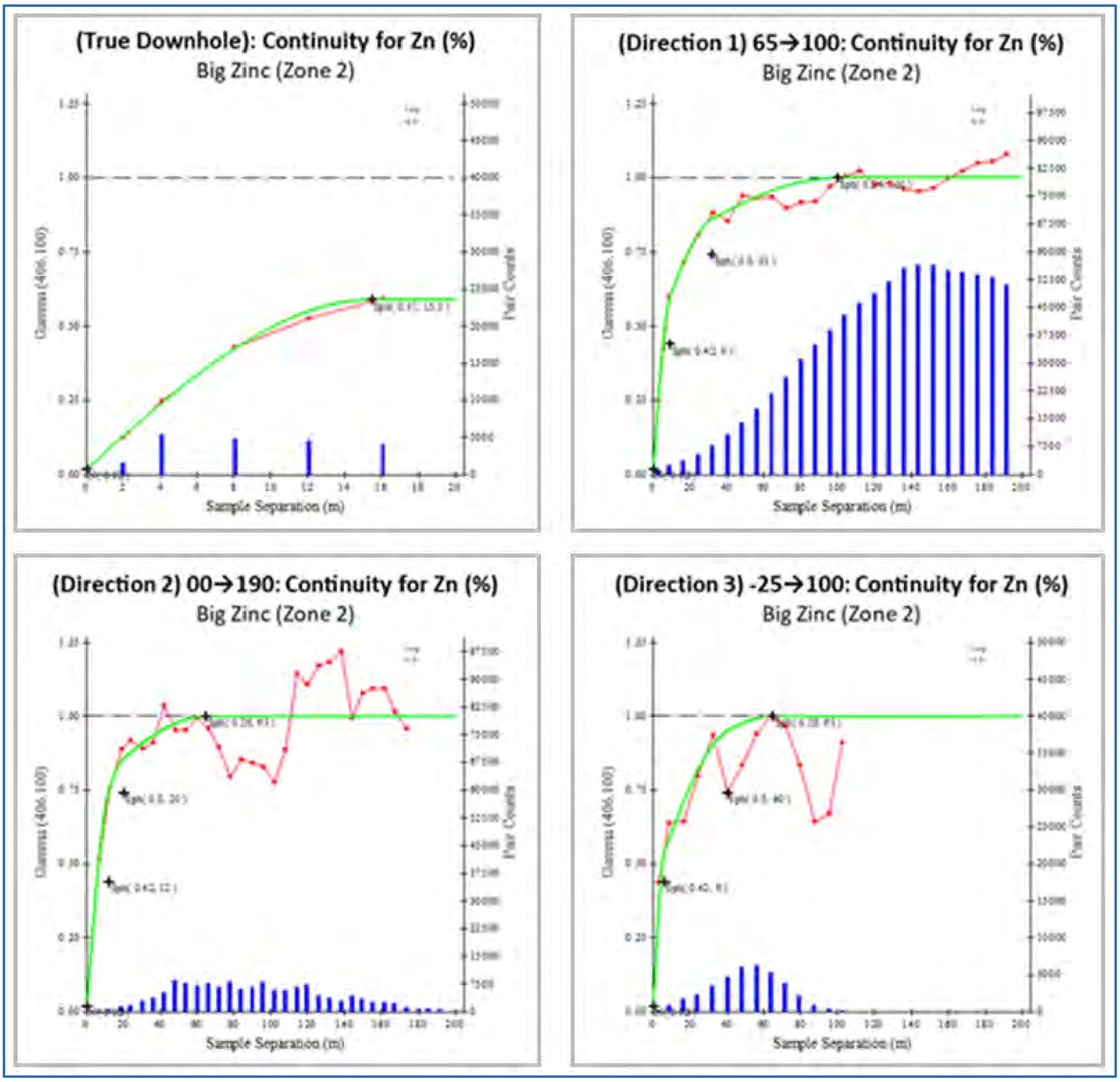

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Table 14.7 Kipushi – Variogram Parameters

Attribute	Transform	Rotation Angle			Nugget Effect (C0)	Range of Structure 1 (R1)			Sill 1 (C1)	Range of Structure 2 (R2)			Sill 2 (C2)	Range of Structure 3 (R3)			Sill 3 (C3)
		1	2	3		1	2	3		1	2	3		1	2	3	
Zone 1 and 8 – Fault Zone																	
Cu %	NS	–65	65	70	0.06	25	25	15	0.30	35	28	15	0.64	–	–	–	–
Pb %	NS	–65	65	70	0.03	15	15	9	0.30	60	60	19	0.67	–	–	–	–
Zn %	NS	–65	65	70	0.02	10	35	4	0.38	60	60	15	0.60	–	–	–	–
S %	NS	–65	65	70	0.01	27	36	5	0.33	30	36	11	0.66	–	–	–	–
As %	NS	–65	65	70	0.01	10	10	9	0.44	35	35	9	0.55	–	–	–	–
Ag g/t	NS	–65	65	70	0.14	7	7	4	0.43	28	18	15	0.43	–	–	–	–
Ge g/t	NS	–65	65	70	0.02	50	50	3	0.28	90	80	28	0.70	–	–	–	–
Co ppm	NS	–65	65	70	0.03	40	40	6	0.27	70	40	14	0.70	–	–	–	–
Cd ppm	NS	–65	65	70	0.02	11	11	6	0.48	90	90	15	0.50	–	–	–	–
Density	None	–65	65	70	0.04	15	15	6	0.40	45	45	9	0.56	–	–	–	–
Iron %	NS	–65	65	70	0.02	20	20	5	0.30	37	37	20	0.68	–	–	–	–
Zone 2 – Big Zinc																	
Cu %	NS	100	115	90	0.02	12	5	5	0.51	50	40	40	0.09	135	120	115	0.38
Pb %	NS	100	115	90	0.02	10	6	5	0.30	58	38	55	0.11	200	45	70	0.57
Zn %	None	100	115	90	0.02	9	12	6	0.43	32	20	40	0.30	100	65	65	0.26
S %	None	100	115	90	0.02	11	14	5	0.37	45	37	35	0.23	70	37	78	0.38
As %	NS	100	115	90	0.03	8	4	6	0.40	32	32	11	0.29	90	55	50	0.28
Ag g/t	NS	100	115	90	0.04	30	20	20	0.47	50	50	50	0.49	–	–	–	–
Ge g/t	NS	100	115	90	0.02	12	12	12	0.47	50	55	30	0.51	–	–	–	–
Co ppm	NS	100	115	90	0.04	12	7	5	0.28	64	33	10	0.32	64	64	42	0.36
Cd ppm	None	100	115	90	0.02	15	12	12	0.55	50	40	43	0.43	–	–	–	–
Density	None	100	115	90	0.04	18	18	10	0.48	65	47	47	0.48	–	–	–	–
Iron %	NS	100	115	90	0.04	10	10	10	0.47	105	90	50	0.49	–	–	–	–
Zone 3 – Southern Zinc																	
Cu %	NS	–65	65	70	0.08	38	40	25	0.92	–	–	–	–	–	–	–	–
Pb %	NS	–65	65	70	0.04	25	16	12	0.96	–	–	–	–	–	–	–	–
Zn %	None	–65	65	70	0.02	6	6	14	0.48	50	32	18	0.5	–	–	–	–
S %	None	–65	65	70	0.14	9	9	6	0.41	45	28	13	0.45	–	–	–	–
As %	NS	–65	65	70	0.06	50	50	20	0.94	–	–	–	–	–	–	–	–
Ag g/t	NS	–65	65	70	0.17	34	40	15	0.83	–	–	–	–	–	–	–	–
Ge g/t	NS	–65	65	70	0.05	10	10	10	0.49	52	52	15	0.46	–	–	–	–
Co ppm	NS	–65	65	70	0.17	33	32	15	0.83	–	–	–	–	–	–	–	–
Cd ppm	NS	–65	65	70	0.02	9	20	15	0.45	80	50	20	0.53	–	–	–	–
Density	None	–65	65	70	0.11	10	10	6	0.45	38	30	15	0.44	–	–	–	–
Iron %	None	–65	65	70	0.17	35	65	10	0.25	105	65	20	0.58	–	–	–	–
Zone 4 – Série Récurrente																	
Cu %	NS	–170	95	–50	0.08	15	12	13	0.42	150	30	30	0.16	150	150	30	0.34
Pb %	NS	–170	95	–50	0.05	16	12	4	0.16	35	25	29	0.11	200	96	80	0.68
Zn %	NS	–170	95	–50	0.05	40	40	25	0.30	155	120	55	0.65	–	–	–	–
S %	NS	–170	95	–50	0.08	135	100	30	0.92	–	–	–	–	–	–	–	–
As %	NS	–170	95	–50	0.04	8	10	5	0.22	50	12	50	0.31	140	125	80	0.43
Ag g/t	NS	–170	95	–50	0.16	15	15	7	0.39	125	125	30	0.28	125	125	52	0.17
Ge g/t	NS	–170	95	–50	0.08	35	48	30	0.45	80	63	40	0.47	–	–	–	–
Co ppm	NS	–170	95	–50	0.34	30	22	7	0.35	130	110	60	0.65	–	–	–	–
Cd ppm	NS	–170	95	–50	0.02	25	20	40	0.33	130	110	60	0.65	–	–	–	–
Density	NS	–170	95	–50	0.02	90	75	70	0.98	–	–	–	–	–	–	–	–
Iron %	NS	–170	95	–50	0.07	130	95	30	0.93	–	–	–	–	–	–	–	–

Attribute	Transform	Rotation Angle			Nugget Effect (C0)	Range of Structure 1 (R1)			Sill 1 (C1)	Range of Structure 2 (R2)			Sill 2 (C2)	Range of Structure 3 (R3)			Sill 3 (C3)
		1	2	3		1	2	3		1	2	3		1	2	3	
Zone 5 and 8 – Série Récurrente Footwall																	
Cu %	NS	–160	85	–40	0.72	50	30	9	0.28	–	–	–	–	–	–	–	–
Pb %	NS	–160	85	–40	0.10	45	30	10	0.90	–	–	–	–	–	–	–	–
Zn %	NS	–160	85	–40	0.13	50	40	9	0.87	–	–	–	–	–	–	–	–
S %	NS	–160	85	–40	0.48	50	45	4	0.52	–	–	–	–	–	–	–	–
As %	NS	–160	85	–40	0.55	25	25	6	0.45	–	–	–	–	–	–	–	–
Ag g/t	NS	–160	85	–40	0.81	30	30	4	0.19	–	–	–	–	–	–	–	–
Ge g/t	NS	–160	85	–40	0.04	50	45	7	0.96	–	–	–	–	–	–	–	–
Co ppm	NS	–160	85	–40	0.60	60	35	6	0.40	–	–	–	–	–	–	–	–
Cd ppm	NS	–160	85	–40	0.19	65	35	11	0.81	–	–	–	–	–	–	–	–
Density	NS	–160	85	–40	0.03	40	35	5	0.97	–	–	–	–	–	–	–	–
Iron %	NS	–160	85	–40	0.64	38	25	3	0.36	–	–	–	–	–	–	–	–
Zone 6 – High-grade Copper Zone Within Big Zinc																	
Cu %	NS	130	95	90	0.02	50	50	6	0.98	–	–	–	–	–	–	–	–
Pb %	NS	130	95	90	0.04	38	38	19	0.96	–	–	–	–	–	–	–	–
Zn %	NS	130	95	90	0.02	44	44	30	0.98	–	–	–	–	–	–	–	–
S %	NS	130	95	90	0.02	40	40	28	0.98	–	–	–	–	–	–	–	–
As %	NS	130	95	90	0.03	40	40	8	0.97	–	–	–	–	–	–	–	–
Ag g/t	NS	130	95	90	0.14	50	50	9	0.86	–	–	–	–	–	–	–	–
Ge g/t	NS	130	95	90	0.07	40	40	8	0.93	–	–	–	–	–	–	–	–
Co ppm	NS	130	95	90	0.07	40	40	8	0.93	–	–	–	–	–	–	–	–
Cd ppm	NS	130	95	90	0.05	36	36	30	0.95	–	–	–	–	–	–	–	–
Density	None	130	95	90	0.04	40	40	30	0.96	–	–	–	–	–	–	–	–
Iron %	None	130	95	90	0.02	43	43	20	0.98	–	–	–	–	–	–	–	–
Zone 7 – Fault Zone Splay																	
Cu %	None	90	90	90	0.03	40	40	10	0.97	–	–	–	–	–	–	–	–
Pb %	None	90	90	90	0.03	40	40	10	0.97	–	–	–	–	–	–	–	–
Zn %	None	90	90	90	0.03	40	40	10	0.97	–	–	–	–	–	–	–	–
S %	None	90	90	90	0.03	40	40	10	0.97	–	–	–	–	–	–	–	–
As %	None	90	90	90	0.03	40	40	10	0.97	–	–	–	–	–	–	–	–
Ag g/t	None	90	90	90	0.03	40	40	10	0.97	–	–	–	–	–	–	–	–
Ge g/t	None	90	90	90	0.03	40	40	10	0.97	–	–	–	–	–	–	–	–
Co ppm	None	90	90	90	0.03	40	40	10	0.97	–	–	–	–	–	–	–	–
Cd ppm	None	90	90	90	0.03	40	40	10	0.97	–	–	–	–	–	–	–	–
Density	None	90	90	90	0.03	40	40	10	0.97	–	–	–	–	–	–	–	–
Iron %	None	90	90	90	0.03	40	40	10	0.97	–	–	–	–	–	–	–	–
Zone 10 – Southern portion of Southern Zinc																	
Cu %	NS	–35	60	70	0.08	38	40	25	0.92	–	–	–	–	–	–	–	–
Pb %	NS	–35	60	70	0.04	25	16	12	0.96	–	–	–	–	–	–	–	–
Zn %	NS	–35	60	70	0.02	6	6	14	0.48	50	32	18	0.50	–	–	–	–
S %	NS	–35	60	70	0.14	9	9	6	0.45	45	28	13	0.45	–	–	–	–
As %	NS	–35	60	70	0.06	50	50	20	0.94	–	–	–	–	–	–	–	–
Ag g/t	NS	–35	60	70	0.17	34	40	15	0.83	–	–	–	–	–	–	–	–
Ge g/t	NS	–35	60	70	0.05	10	10	10	0.49	52	52	15	0.46	–	–	–	–
Co ppm	NS	–35	60	70	0.17	33	32	15	0.83	–	–	–	–	–	–	–	–
Cd ppm	NS	–35	60	70	0.02	9	20	15	0.45	80	50	20	0.53	–	–	–	–
Density	NS	–35	60	70	0.11	10	10	6	0.45	38	30	15	0.44	–	–	–	–
Iron %	NS	–35	60	70	0.17	35	65	10	0.25	105	65	20	0.58	–	–	–	–

All variograms are rotated on the Datamine Z-X-Z rotation logic.

14.5.2 Indicator Variograms

The mineralisation at Kipushi, in particular the Big Zinc Zone, consists of extensive massive sulfide zones with internal pods of low-grade material. It would be in-optimal to dilute the high-grade massive sulfide zones with lower-grades from low-grade pods within these zones. Some of the low-grade zones are caused by zero grades being applied to un-sampled intervals of the Gécamines drillholes. An indicator approach was used to discriminate between the high and low-grade zones. The Indicator approach was only necessary for the Fault Zone, Série Récurrente, Big Zinc, and Southern Zinc.

Indicator variograms were calculated using the 2 m sample composites and modelled at a threshold of 5% Zn for the zinc-rich zones (Zone 2 and 3) and 0.5% Cu for the copper-rich zones (Zone 1 and Zone 4). The indicator variograms were modelled in three directions. The variogram models for Zone 2 and Zone 4 are robust and are informed by a reasonable number of experimental data, although the indicator variograms for Zone 1 and Zone 3 are poorly structured. The indicator variograms are shown in Figure 14.7, Figure 14.8 and Figure 14.9 and the indicator variogram parameters are shown in Table 14.8.

Figure 14.7 Zone 2 – 5% Zinc Indicator Variograms

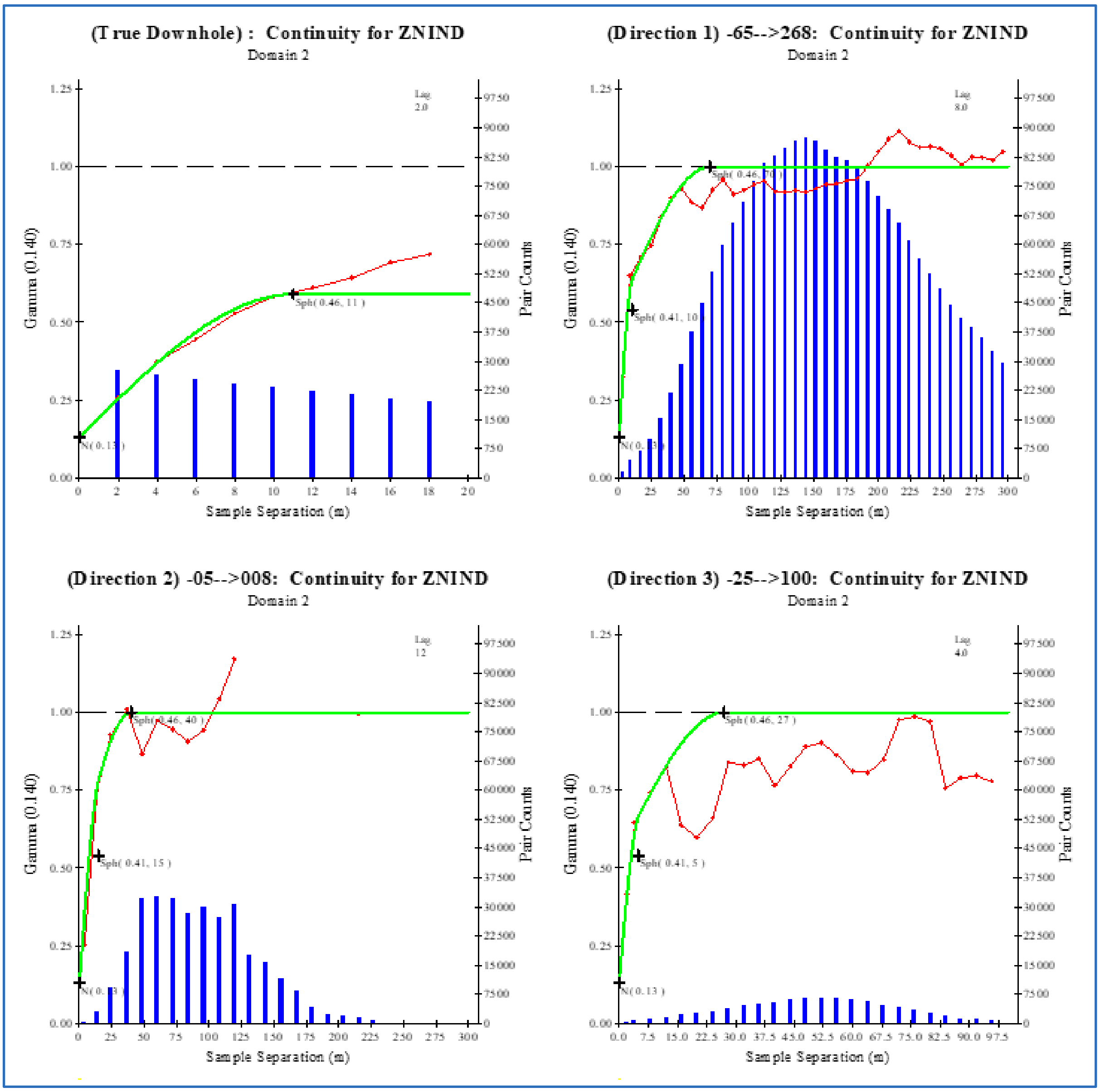

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Figure 14.8 Zone 3 – 5% Zinc Indicator Variograms

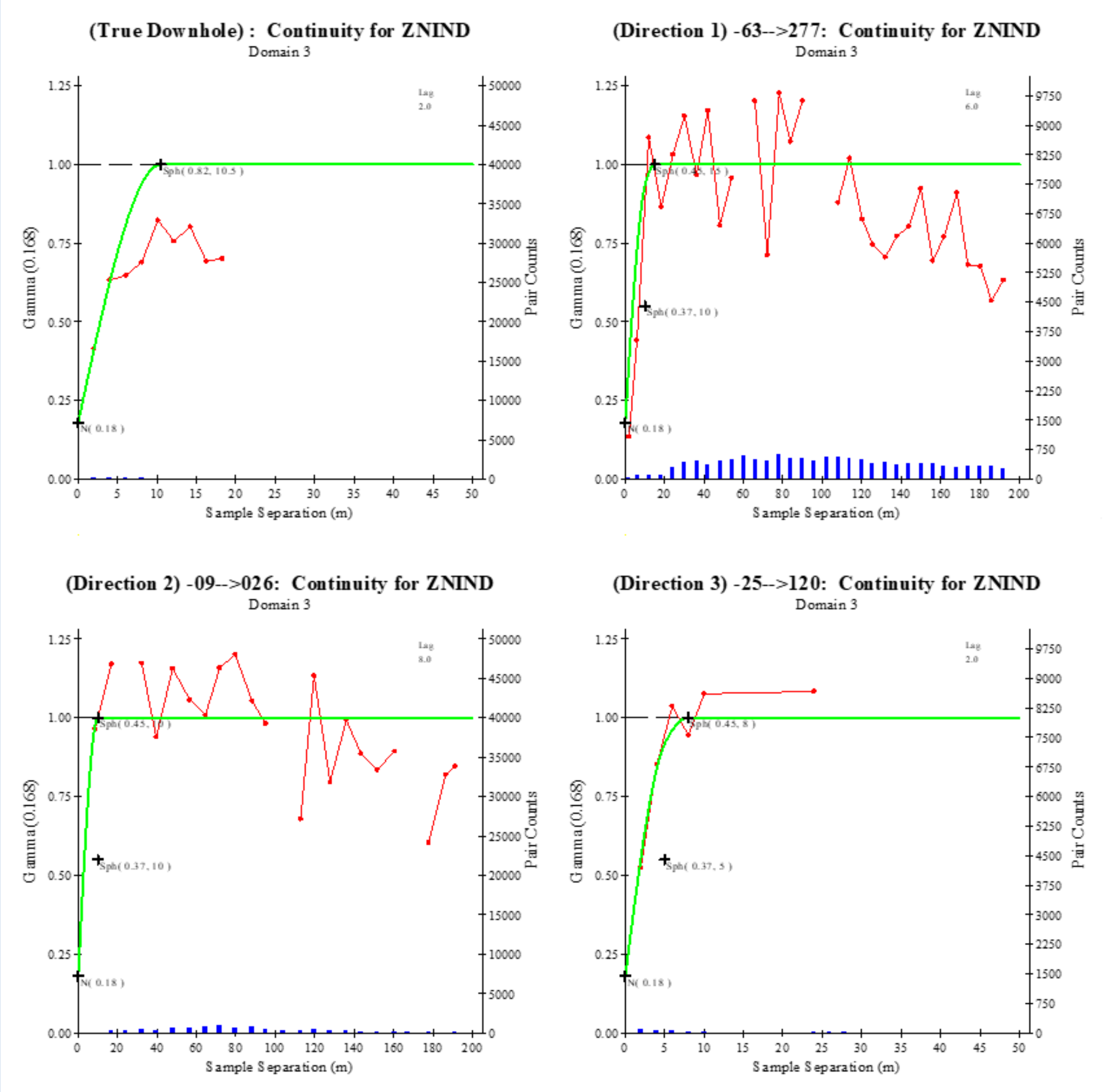

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Figure 14.9 Zone 4 – 0.5% Copper Indicator Variograms

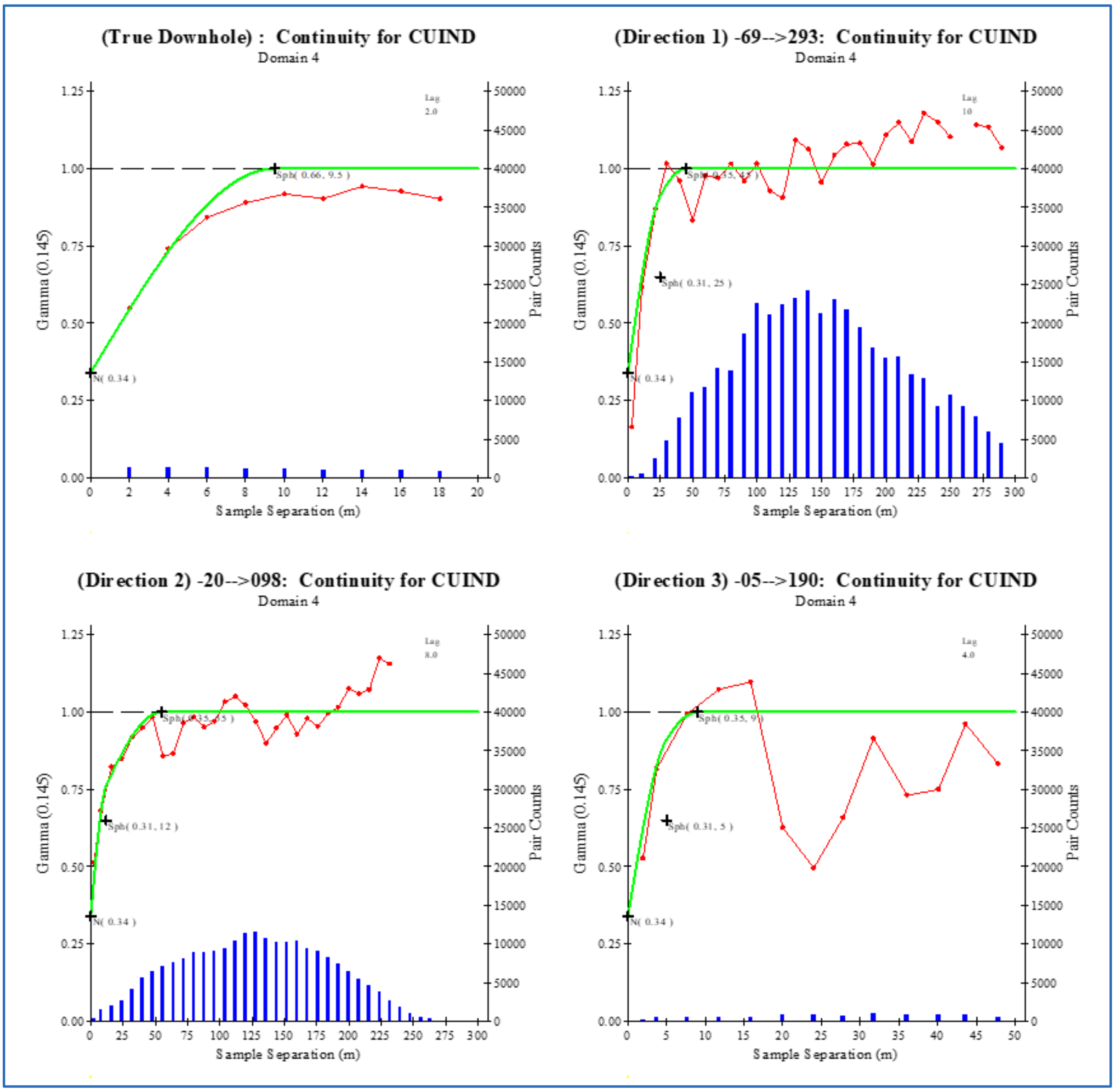

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Table 14.8 Kipushi – Indicator Variogram Parameters

Attribute	Transform	Rotation Angle			Rotation Axis			Nugget Effect (C0)	Range of Structure 1 (R1)			Sill 1 (C1)	Range of Structure 2 (R2)			Sill 2 (C2)
		1	2	3	1	2	3		1	2	3		1	2	3	
Fault Zone																
Cu Indicator (0.5%)	None	115	115	–75	Z	X	Z	0.23	25	25	16	0.40	45	32	17	0.37
Big Zinc																
Zinc Indicator (0.5%)	None	100	115	–85	Z	X	Z	0.13	10	15	5	0.41	70	40	27	0.46
Southern Zinc																
Zn Indicator (5%)	None	120	115	–80	Z	X	Z	0.18	10	10	5	0.37	15	10	8	0.45
Série Récurrente																
Cu Indicator (0.5%)	None	–170	95	–70	Z	X	Z	0.34	25	12	5	0.31	45	55	9	0.35

14.6 Block Modelling

The wireframes were filled with cells with a dimension of 5 m X x 5 m Y x 5 m Z, which is one third of the 15 m spaced drilling sections. The drilling was at various inclinations and the grade trends vary between the zones, so an equidimensional block size was considered appropriate.

The parent cells were sub-celled to a minimum of 0.5 m X x 0.5 m Y x 0.5 m Z in order to best fill the irregular shapes of the mineralised bodies.

The 10 different zone wireframes were filled separately and the blocks were coded with the respective zone code.

The block model volume was compared to the wireframe volume and differences of less than 0.5% were found between the two, indicating that the wireframes were appropriately filled with block model cells.

14.7 Estimation

14.7.1 Indicator Estimation

In order to retain the high-grades in the massive zones and the low-grades in the isolated internal low-grade zones without smoothing the grades between them, an indicator approach was used to discriminate between them. The probability of a model cell being above or below a 0.5% Cu or 5% Zn threshold for the copper-rich and zinc-rich domains respectively was estimated using the 2 m composite data transformed to indicators, with ‘1’ being above the threshold value and ‘0’ being below. Ordinary kriging of the indicators into parent cells using the indicator variograms (Section 14.5.2) was carried out. The parameters used for the indicator estimation are shown in Table 14.9. These were aligned with the direction and distance of continuity as implied by the indicator variograms. Should an estimate not be achieved by selecting sufficient composites in the first search, the search was expanded until four composites were selected.

The Indicator approach was only necessary for the Fault Zone, Série Récurrente, Big Zinc, and Southern Zinc.

Table 14.9 Kipushi – Indicator Search Parameters

Attribute	Search Angle			Rotation Axis			Search Distance			Number of Composites		Second Search Multiplier	Number of Composites		Third Search Multiplier	Number of Composites	
	1	2	3	1	2	3	1	2	3	Min.	Max.		Min.	Max.		Min.	Max.
Fault Zone (Zone 1)																	
Cu Indicator (0.5%)	110	115	–60	Z	X	Z	100	75	20	4	8	1.5	4	4	10	4	4
Big Zinc (Zone 2)																	
Zinc Indicator (0.5%)	110	115	90	Z	X	Z	160	60	60	4	8	1.5	4	4	10	4	4
Southern Zinc (Zone 3)																	
Zinc Indicator (0.5%)	120	110	90	Z	X	Z	160	60	60	4	8	1.5	4	4	10	4	4
Série Récurrente (Zone 4)																	
Cu Indicator (0.5%)	–170	90	50	Z	X	Z	80	80	40	4	8	1.5	4	4	10	4	4

14.7.2 Grade Estimation

Each of the elements and density were estimated using ordinary kriging by estimating into parent cells.

The indicator estimates were carried out on the major element for each zone (copper for Zones 1 and 4 and zinc for Zones 2 and 3) and those closely related to them so that the indicator approach was applied to the following attributes:

- Zones 1 and 4 – copper, sulfur, iron, and density.
- Zones 2 and 3 – zinc, cadmium, sulfur, iron, and density.

Each cell was estimated twice; an estimate using the below threshold data and an estimate using the above threshold data. The same search parameters and variograms were used to estimate the above and below threshold values. The two estimates were then combined based on the proportion of above or below threshold as determined by the indicator kriging.

The other attributes and zones estimated using ordinary kriging without indicators.

The search parameters used are shown in Table 14.10. A different search distance was allowed for each element, as the different elements tend to behave independently of each other. This is with the exception of cadmium and zinc, which are closely related, and the search parameter for zinc was applied to cadmium to ensure the relationship between these elements was preserved in the estimate.

The search parameters are based on the variogram ranges and anisotropy. The first search distance being the same as the total variogram range and the second search being 1.5 times the variogram range. A third search that sources a minimum of five and maximum of 10 samples was used. This is a greatly expanded search designed to achieve estimates approaching the local mean. A maximum of four composites from a single drillhole were allowed to estimate a cell in order to ensure that each cell was estimated using more than one drillhole. Any cells that were not estimated were assigned the domain average values.

The lead, arsenic, silver, germanium, and cobalt distributions are characterised by small numbers of high-grade values within dominantly low-grade populations. The high-grades tend to occur in clusters. In order to retain the high-grade values locally, without smearing of the values throughout their respective estimation domains, a restricted omnidirectional search of 7 m was applied on the data during interpolation without capping applied. This allows the high-grades to influence only the block in which they occur and the immediate surrounding blocks. The estimates using the uncapped data replaced the estimates using the capped data. This technique honours areas of higher grade with short continuity and does not allow the higher-grades to influence areas that dominantly contain low or background level grade.

No arsenic, silver, germanium, and cobalt assays were available for Zone 10 and the mean capped grades for Zone 1 were applied to it.

14.7.2.1 Boundary Conditions

Each domain was estimated only using the drillhole data within it (hard boundaries).

Table 14.10 Kipushi – Search Parameters

Attribute	Search Angle			Rotation Axis			Search Distance			Number of Composites		Second Search Multiplier	Number of Composites		Third Search Multiplier	Number of Composites	
	1	2	3	1	2	3	1	2	3	Min.	Max.		Min.	Max.		Min.	Max.
Fault Zone (Zone 1 and Zone 9)																	
Cu %	–65	65	70	Z	X	Z	35	28	15	6	12	1.5	6	12	100	5	10
Pb g/t	–65	65	70	Z	X	Z	60	60	19	6	12	1.5	6	12	100	5	10
Zn %	–65	65	70	Z	X	Z	60	60	15	6	12	1.5	6	12	100	5	10
S %	–65	65	70	Z	X	Z	30	36	11	6	12	1.5	6	12	100	5	10
As %	–65	65	70	Z	X	Z	35	35	9	6	12	1.5	6	12	100	5	10
Ag g/t	–65	65	70	Z	X	Z	28	18	15	6	12	1.5	6	12	100	5	10
Ge g/t	–65	65	70	Z	X	Z	90	80	28	6	12	1.5	6	12	100	5	10
Co ppm	–65	65	70	Z	X	Z	70	40	14	6	12	1.5	6	12	100	5	10
Cd ppm	–65	65	70	Z	X	Z	60	60	15	6	12	1.5	6	12	100	5	10
Density	–65	65	70	Z	X	Z	45	45	9	6	12	1.5	6	12	100	5	10
Fe %	–65	65	70	Z	X	Z	37	37	20	6	12	1.5	6	12	100	5	10
Big Zinc (Zone 2)																	
Cu %	100	115	90	Z	X	Z	135	120	115	6	12	1.5	6	12	100	5	10
Pb g/t	100	115	90	Z	X	Z	200	45	70	6	12	1.5	6	12	100	5	10
Zn %	100	115	90	Z	X	Z	100	65	35	6	12	1.5	6	12	100	5	10
S %	100	115	90	Z	X	Z	70	37	78	6	12	1.5	6	12	100	5	10
As %	100	115	90	Z	X	Z	90	55	50	6	12	1.5	6	12	100	5	10
Ag g/t	100	115	90	Z	X	Z	50	50	50	6	12	1.5	6	12	100	5	10
Ge g/t	100	115	90	Z	X	Z	50	55	30	6	12	1.5	6	12	100	5	10
Co ppm	100	115	90	Z	X	Z	64	64	32	6	12	1.5	6	12	100	5	10
Cd ppm	100	115	90	Z	X	Z	100	65	65	6	12	1.5	6	12	100	5	10
Density	100	115	90	Z	X	Z	65	47	47	6	12	1.5	6	12	100	5	10
Fe %	100	115	90	Z	X	Z	105	90	50	6	12	1.5	6	12	100	5	10
Southern Zinc (Zone 3)																	
Cu %	–60	65	70	Z	X	Z	80	70	10	6	12	1.5	6	12	100	5	10
Pb g/t	–60	65	70	Z	X	Z	170	40	23	6	12	1.5	6	12	100	5	10
Zn %	–60	65	70	Z	X	Z	38	40	25	6	12	1.5	6	12	100	5	10
S %	–60	65	70	Z	X	Z	25	16	12	6	12	1.5	6	12	100	5	10
As %	–60	65	70	Z	X	Z	50	32	18	6	12	1.5	6	12	100	5	10
Ag g/t	–60	65	70	Z	X	Z	45	28	13	6	12	1.5	6	12	100	5	10
Ge g/t	–60	65	70	Z	X	Z	50	50	20	6	12	1.5	6	12	100	5	10
Co ppm	–60	65	70	Z	X	Z	34	40	15	6	12	1.5	6	12	100	5	10
Cd ppm	–60	65	70	Z	X	Z	52	52	15	6	12	1.5	6	12	100	5	10
Density	–60	65	70	Z	X	Z	33	32	15	6	12	1.5	6	12	100	5	10
Fe %	–60	65	70	Z	X	Z	105	65	20	6	12	1.5	6	12	100	5	10
Série Récurrente (Zone 4)																	
Cu %	–170	95	50	Z	X	Z	150	150	30	6	12	1.5	6	12	100	5	10
Pb g/t	–170	95	50	Z	X	Z	200	96	80	6	12	1.5	6	12	100	5	10
Zn %	–170	95	50	Z	X	Z	155	120	55	6	12	1.5	6	12	100	5	10
S %	–170	95	50	Z	X	Z	135	100	30	6	12	1.5	6	12	100	5	10
As %	–170	95	50	Z	X	Z	140	125	80	6	12	1.5	6	12	100	5	10
Ag g/t	–170	95	50	Z	X	Z	125	125	52	6	12	1.5	6	12	100	5	10
Ge g/t	–170	95	50	Z	X	Z	80	63	40	6	12	1.5	6	12	100	5	10
Co ppm	–170	95	50	Z	X	Z	58	25	17	6	12	1.5	6	12	100	5	10
Cd ppm	–170	95	50	Z	X	Z	155	120	55	6	12	1.5	6	12	100	5	10
Density	–170	95	50	Z	X	Z	90	75	40	6	12	1.5	6	12	100	5	10
Fe %	Fe %	–170	95	50	Z	X	Z	180	70	39	6	12	1.5	6	12	100	5
High–grade Zone in Série Récurrente (Zone 5 and Zone 8)																	
Cu %	–160	85	–40	Z	X	Z	50	30	9	6	12	1.5	6	12	100	5	10
Pb g/t	–160	85	–40	Z	X	Z	45	30	10	6	12	1.5	6	12	100	5	10

Attribute	Search Angle			Rotation Axis			Search Distance			Number of Composites		Second Search Multiplier	Number of Composites		Third Search Multiplier	Number of Composites	
	1	2	3	1	2	3	1	2	3	Min.	Max.		Min.	Max.		Min.	Max.
Zn %	–160	85	–40	Z	X	Z	50	40	9	6	12	1.5	6	12	100	5	10
S %	–160	85	–40	Z	X	Z	50	45	4	6	12	1.5	6	12	100	5	10
As %	–160	85	–40	Z	X	Z	25	25	6	6	12	1.5	6	12	100	5	10
Ag g/t	–160	85	–40	Z	X	Z	35	30	4	6	12	1.5	6	12	100	5	10
Ge g/t	–160	85	–40	Z	X	Z	50	45	7	6	12	1.5	6	12	100	5	10
Co ppm	–160	85	–40	Z	X	Z	60	35	6	6	12	1.5	6	12	100	5	10
Cd ppm	–160	85	–40	Z	X	Z	50	40	9	6	12	1.5	6	12	100	5	10
Density	–160	85	–40	Z	X	Z	40	35	5	6	12	1.5	6	12	100	5	10
Fe %	–160	85	–40	Z	X	Z	38	25	3	6	12	1.5	6	12	100	5	10
Copper Rich Zone in Big Zinc (Zone 6)																	
Cu %	130	95	90	Z	X	Z	50	50	6	6	12	1.5	6	12	100	5	10
Pb g/t	130	95	90	Z	X	Z	38	38	19	6	12	1.5	6	12	100	5	10
Zn %	130	95	90	Z	X	Z	44	44	30	6	12	1.5	6	12	100	5	10
S %	130	95	90	Z	X	Z	40	40	28	6	12	1.5	6	12	100	5	10
As %	130	95	90	Z	X	Z	40	40	8	6	12	1.5	6	12	100	5	10
Ag g/t	130	95	90	Z	X	Z	50	50	9	6	12	1.5	6	12	100	5	10
Ge g/t	130	95	90	Z	X	Z	40	40	8	6	12	1.5	6	12	100	5	10
Co ppm	130	95	90	Z	X	Z	65	65	28	6	12	1.5	6	12	100	5	10
Cd ppm	130	95	90	Z	X	Z	44	44	30	6	12	1.5	6	12	100	5	10
Density	130	95	90	Z	X	Z	40	40	30	6	12	1.5	6	12	100	5	10
Fe %	130	95	90	Z	X	Z	43	43	20	6	12	1.5	6	12	100	5	10
Fault Splay Zone (Zone 7)																	
Cu %	90	90	90	Z	X	Z	40	40	10	6	12	1.5	6	12	100	5	10
Pb g/t	90	90	90	Z	X	Z	40	40	10	6	12	1.5	6	12	100	5	10
Zn %	90	90	90	Z	X	Z	40	40	10	6	12	1.5	6	12	100	5	10
S %	90	90	90	Z	X	Z	40	40	10	6	12	1.5	6	12	100	5	10
As %	90	90	90	Z	X	Z	40	40	10	6	12	1.5	6	12	100	5	10
Ag g/t	90	90	90	Z	X	Z	40	40	10	6	12	1.5	6	12	100	5	10
Ge g/t	90	90	90	Z	X	Z	40	40	10	6	12	1.5	6	12	100	5	10
Co ppm	90	90	90	Z	X	Z	40	40	10	6	12	1.5	6	12	100	5	10
Cd ppm	90	90	90	Z	X	Z	40	40	10	6	12	1.5	6	12	100	5	10
Density	90	90	90	Z	X	Z	40	40	10	6	12	1.5	6	12	100	5	10
Fe %	90	90	90	Z	X	Z	40	40	10	6	12	1.5	6	12	100	5	10
Copper Rich Zones in Southern Zinc (Zone 10)																	
Cu %	–35	60	70	Z	X	Z	38	40	25	6	12	1.5	6	12	100	5	10
Pb g/t	–35	60	70	Z	X	Z	25	16	12	6	12	1.5	6	12	100	5	10
Zn %	–35	60	70	Z	X	Z	50	32	18	6	12	1.5	6	12	100	5	10
S %	–35	60	70	Z	X	Z	45	28	13	6	12	1.5	6	12	100	5	10
As %	–35	60	70	Z	X	Z	50	50	20	6	12	1.5	6	12	100	5	10
Ag g/t	–35	60	70	Z	X	Z	34	40	15	6	12	1.5	6	12	100	5	10
Ge g/t	–35	60	70	Z	X	Z	52	52	15	6	12	1.5	6	12	100	5	10
Co ppm	–35	60	70	Z	X	Z	33	32	15	6	12	1.5	6	12	100	5	10
Cd ppm	–35	60	70	Z	X	Z	50	32	18	6	12	1.5	6	12	100	5	10
Density	–35	60	70	Z	X	Z	38	30	15	6	12	1.5	6	12	100	5	10
Fe %	–35	60	70	Z	X	Z	105	65	20	6	12	1.5	6	12	100	5	10

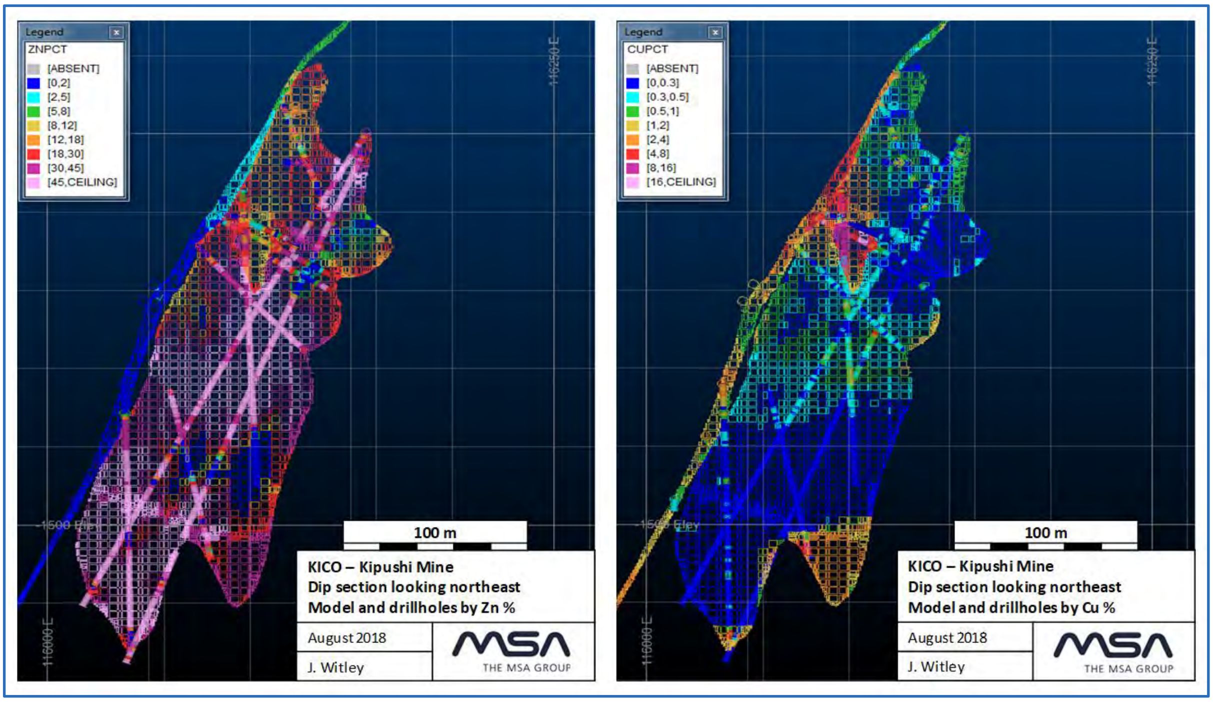
14.8 Validation of the Estimates

The models were validated by:

- Visual examination of the input data against the block model estimates,
- Sectional validation, and
- Comparison of the input data statistics against the model statistics.

The block model was examined visually in sections to ensure that the drillhole grades were locally well represented by the model. It was found that the model validated reasonably well against the data. A section showing the block model and drillholes is shown in Figure 14.10.

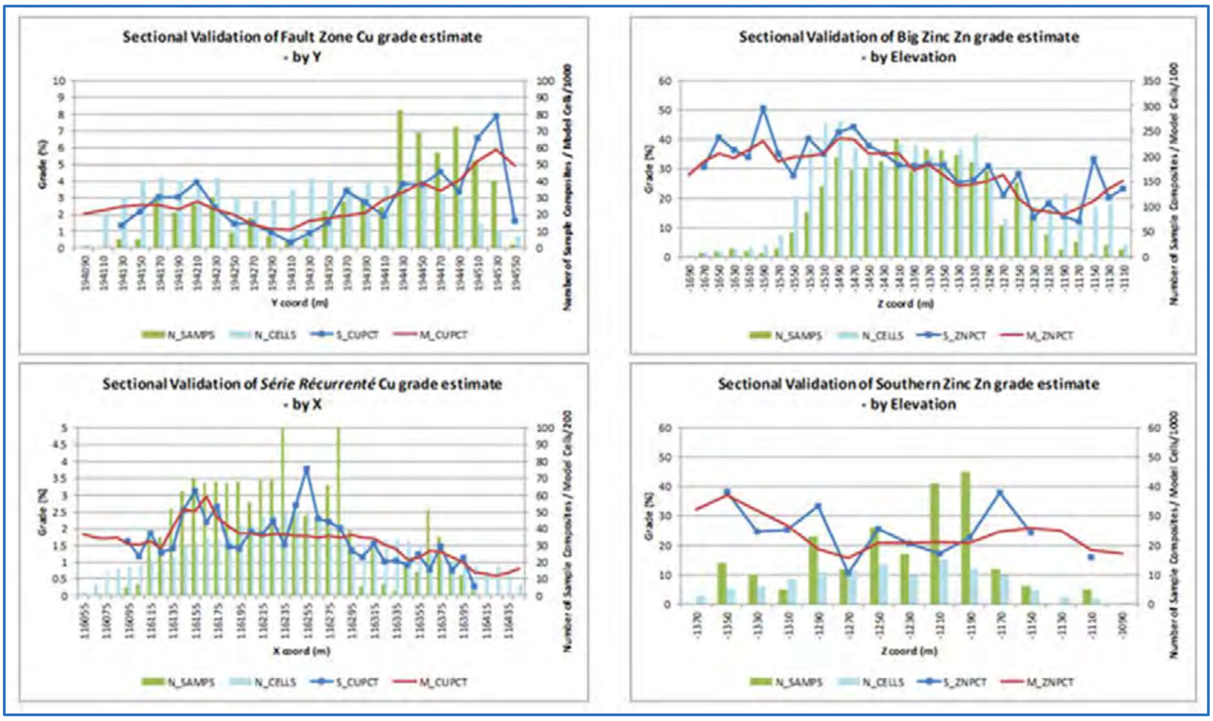
Figure 14.10 Section Through Big Zinc and Fault Zone Block Model and Drillhole Data Illustrating Correlation between Model and Data



MSA, 2018
Shaded by Zinc (Left) and Copper (Right)

Sectional validation plots were constructed for each major element and each zone. The sectional validation plots compare the average grades of the block model against the input data along a number of corridors in various directions through the deposit. Samples of the sectional validation plots are shown in Figure 14.11. These show that the estimates retain the local grade trends across the deposit.

Figure 14.11 Sectional Validation Plots



MSA, 2018

As a further check, the de-clustered drillhole composite mean grades were compared with the model grade. The model and the data averages compare reasonably well for most variables. Those that did not compare within reasonable limits were examined further. No consistent biases were found, and the differences were all explained by the arrangement of the data relative to the volume of the model and are of no concern. For the elements that were estimated using the restricted uncapped search (lead, arsenic, silver, germanium, and cobalt) higher discrepancies between the capped mean and model mean tended to occur. The more significant discrepancies between the capped mean and model mean are explained as follows:

- The Zone 1 germanium model grade is 62.3% higher than the capped mean and is 36% higher than the uncapped mean. Only the KICO drillholes were assayed for germanium and a large proportion of the model was outside of the KICO drilling area. The data on the fringes of the KICO drilling area, which are higher than the data mean, have been extrapolated to the south-west.
- The lead estimates for several zones are significantly higher than the capped mean but lower than the uncapped mean. This is a function of the restrictive search on the uncapped data.
- The germanium model grade for Zone 7 is 33.5% higher than the data mean. The data for this model is sparse and irregularly spaced and the estimate is, therefore, very susceptible to the data arrangement. Relatively high-grades have been extrapolated into a large poorly informed area to the north. This is also the case for cadmium.

- The zinc model grade for Zone 8 is 33.1% higher than the mean data grade. For cadmium the model grade is 34.3% higher than the mean data grade. The lower-grade data tends to occur on the edges of the model and, therefore, have less influence than the higher-grade data that occur towards the centre of the model. The model was examined in detail visually and with sectional validation plots and no issues were found.

14.9 Mineral Resource Classification

Classification of the Kipushi Mineral Resource was based on confidence in the data, confidence in the geological model, grade continuity and variability, and the frequency of the drilling data. The main considerations in the classification of the Kipushi Mineral Resource are as follows:

- The data were collected by KICO and Gécamines. The KICO data have been collected using current industry standard principles; however, the quality of the Gécamines data is less certain. KICO has endeavoured to verify the Gécamines data by a programme of re-sampling and twin drilling in the Big Zinc and portions of the Fault Zone which yielded reasonable comparisons.
- The Gécamines data are incomplete in several aspects; notably not all of the elements of interest were analysed, and the sampling was selective in some of the drillholes. A rigorous validation exercise was completed that resulted in many of the Gécamines holes being rejected for use in the grade estimate.
- Areas of the Fault Zone, Série Récurrente and the southern portion of the Southern Zinc are only informed by Gécamines drillholes. The Big Zinc has been well drilled by KICO as well as a portion of the Série Récurrente and the Fault Zone.
- The geological framework of the Mineral Resource is well understood as are the controls to the mineralisation.
- The Mineral Resource has been densely drilled on sections spaced 15 m apart, although areas of the Série Récurrente and down dip areas of the Fault Zone are less well drilled.
- Variogram ranges are well in excess of the drillhole spacing.
- The grade model validates reasonably well, although suffers from a lack of data for several elements notably silver, germanium, and cobalt as these assays were not available in the database constructed from the Gécamines data.
- Kipushi has an extensive mining history and the continuity of the mineralised bodies has been established through mining.

Given the aforementioned factors the Kipushi Mineral Resource was classified using the following criteria:

- One area of the Big Zinc Zone and adjacent Fault Zone was classified as Measured. The spacing of the KICO drillholes in this area is less than 20 m and there is high confidence in the interpretation of the mineralised extents.

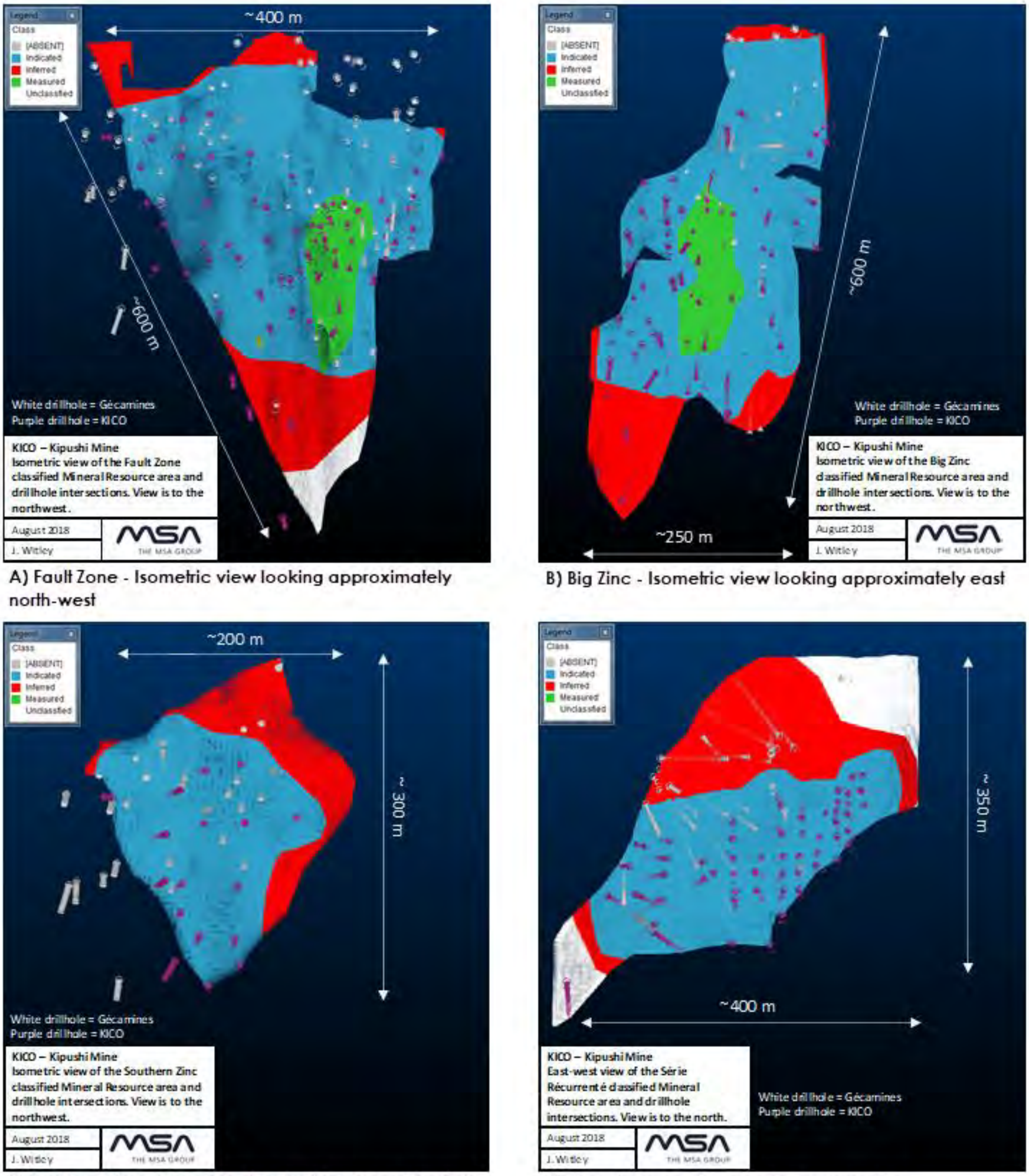
- Where informed predominantly by KICO drilling, and with a drillhole spacing of closer than 50 m, the Mineral Resource was classified as Indicated. This applies to the majority of the Big Zinc, the Fault Zone in the vicinity of the Big Zinc and Southern Zinc, the northern and central portions of the Southern Zinc and an area of the Série Récurrente. Consideration of the proximity to the areas of historical mining was made, as in general these will be of lower risk.
- For areas of the Mineral Resource predominantly informed by Gécamines drillholes, the Mineral Resource was classified as Inferred. This applies to the southern portion of the Southern Zinc and areas of the Fault Zone and Série Récurrente.
- The Fault Zone Splay was classified as Inferred. This zone is informed by six KICO drillholes, many of which are drilled at a close angle to the plane of the mineralisation. Grades in this area are variable and the interpretation of the mineralised extents is tenuous.
- Extrapolation of the Big Zinc was limited to a maximum of 15 m, the complex shape of the deposit negated against greater extrapolation with any confidence. The Fault Zone and Série Récurrente are highly continuous and the down dip extent was limited to 50 m from the drillhole intersections.

The classified areas for the Big Zinc, Fault Zone, Southern Zinc and Série Récurrente are shown in Figure 14.12.

To the best of the Qualified Person's knowledge there is no environmental, permitting, legal, tax, socio-political, marketing, or other relevant issues which may materially affect the Mineral Resource estimate as reported in the Kipushi 2019 Resource Update, aside from those mentioned in Section 4 of this report.

The Mineral Resources could be affected by further infill and exploration drilling, which may result in increases or decreases in subsequent Mineral Resource estimates. Inferred Mineral Resources are considered to be high risk estimates that may change significantly with additional data. It cannot be assumed that all or part of an Inferred Mineral Resource will necessarily be upgraded to an Indicated Mineral Resource as a result of continued exploration. The Mineral Resources may also be affected by subsequent assessments of mining, environmental, processing, permitting, taxation, socio-economic, and other factors.

Figure 14.12 Mineral Resource Classification



A) Fault Zone - Isometric view looking approximately north-west

B) Big Zinc - Isometric view looking approximately east

C) Southern Zinc - Isometric view looking approximately north-west

D) Série Récurrenté - Isometric view looking approximately south-east

MSA, 2018; Only drillholes used for estimation shown. Only area in DRC shown.

14.10 Depletion of the Mineral Resource

The grade model includes areas that have previously been mined by Gécamines and an area to the south-west inside Zambia.

14.10.1 Mined out Areas

Mined out areas were supplied by KICO. These were simplified into cohesive areas, so that isolated remnants were not included in the Mineral Resource estimate, and then used for depletion of the model. In addition, the entire model above 1,150 mL was removed, extensive mining having taken place in that region. There is potential for additional Mineral Resources to exist above 1,150 mL but this will require investigation in terms of mineralisation remaining and reasonable prospects for eventual economic extraction of the remnant areas.

14.10.2 Zambia-DRC Border

The mineralisation at Kipushi straddles the Democratic Republic of Congo (DRC)-Zambia border, however, the exact position of the border is uncertain at Kipushi, as there is currently no official surveyed border line available for the area.

KICO commissioned a professional land surveyor (Mr DJ Cochran – Pr.MS, PLATO, SAGI of CAD Mapping Aerial Surveyors based in Tshwane, South Africa) to determine the position of the border as accurately as possible (Cochran, 2015).

Mr Cochran located the position of four of the original border beacons (probably from the early 1930's) and surveyed them using high precision GNSS post processing systems (on ITRF2008/WGS84). Together with information obtained by interviewing local inhabitants and from the Zambian Department of Survey and Lands in Lusaka, a pragmatic border line was interpreted (Figure 14.13). Mr Cochran is confident that the pragmatic border line best represents the most likely border line. The interpreted border line generally fits to the surveyed beacons to within ± 0.5 m and follows the general trend of the watershed in the area.

Figure 14.13 Google Earth Image Showing Position of DRC-Zambia Border

The border from Google Earth is shown in yellow and the pragmatic border line in green.
Source- Google Earth and Cochran, 2015.

The pragmatic border line was projected vertically to the Kipushi mineralisation models and all modelled mineralisation on the Zambian side of the border line was removed from the Mineral Resource estimate.

14.11 Mineral Resource Statement

The Mineral Resource was estimated using The Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Best Practice Guidelines and is reported in accordance with the 2014 CIM Definition Standards, which have been incorporated by reference into National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101). The Mineral Resource is classified into the Measured, Indicated, and Inferred categories as shown in Table 14.11 for the predominantly zinc-rich bodies and in Table 14.12 for the predominantly copper-rich bodies.

The Measured, Indicated, and Inferred Mineral Resource for the zinc-rich bodies has been tabulated using a number of cut-off grades as shown in Table 14.13 and Table 14.14 respectively and Table 14.15 and Table 14.16 for the copper-rich bodies.

For the zinc-rich zones the Mineral Resource is reported at a base case cut-off grade of 7.0% Zn, and the copper-rich zones at a base case cut-off grade of 1.5% Cu. Given the considerable revenue which will be obtained from the additional metals in each zone, MSA considers that mineralisation at these cut-off grades will satisfy reasonable prospects for economic extraction.

It should be noted that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability and the economic parameters used to assess the potential for economic extraction is not an attempt to estimate Mineral Reserves.

Table 14.11 Kipushi – Zinc-Rich Mineral Resource at 7% Zn Cut-off Grade, 14 June 2018

Zone	Category	Tonnes (Mt)	Zn (%)	Cu (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
Big Zinc	Measured	3.65	39.87	0.65	0.35	18	18	56
	Indicated	7.25	34.36	0.62	1.29	19	12	53
	Inferred	0.98	35.32	1.18	0.09	8	15	62
Southern Zinc	Indicated	0.88	24.52	2.97	1.95	75	6	188
	Inferred	0.16	24.37	1.64	1.20	38	6	61
Total	Measured	3.65	39.87	0.65	0.35	18	18	56
	Indicated	8.13	33.30	0.87	1.36	25	11	68
	Measured and Indicated	11.78	35.34	0.80	1.05	23	13	64
	Inferred	1.14	33.77	1.24	0.24	12	14	62

Contained Metal Quantities								
Zone	Category	Tonnes (Mt)	Zn (Mlb)	Cu (Mlb)	Pb (Mlb)	Ag (Moz)	Co (Mlb)	Ge (Moz)
Big Zinc	Measured	3.65	3,210.6	52.3	27.8	2.06	0.14	6.60
	Indicated	7.25	5,489.0	98.7	206.6	4.48	0.19	12.43
	Inferred	0.98	764.0	25.5	1.9	0.26	0.03	1.96
Southern Zinc	Indicated	0.88	476.5	57.6	37.8	2.11	0.01	5.34
	Inferred	0.16	86.7	5.8	4.3	0.20	0.00	0.32
Total	Measured	3.65	3,210.6	52.3	27.8	2.06	0.14	6.60
	Indicated	8.13	5,965.5	156.4	244.4	6.59	0.20	17.77
	Measured and Indicated	11.78	9,176.0	208.6	272.2	8.65	0.34	24.36
	Inferred	1.14	850.7	31.3	6.2	0.46	0.04	2.28

1. All tabulated data has been rounded and as a result minor computational errors may occur.
2. Mineral Resources are not Mineral Reserves which have no demonstrated economic viability.
3. The Mineral Resource is reported as the total in-situ Mineral Resource and on a 100% project basis, exclusive of Mineral Reserves. Ivanhoe holds an indirect 68% interest in the Project.
4. Metal quantities are reported in multiples of Troy Ounces or Avoirdupois Pounds.
5. The cut-off grade calculation was based on the following assumptions: zinc price of $1.00/lb, mining cost of $50/t, processing cost of $10/t, G&A and holding cost of $10/t, transport of 55% Zn concentrate at $210/t, 90% zinc recovery and 85% payable zinc.

Table 14.12 Kipushi – Copper-Rich Mineral Resource at 1.5% Cu Cut-off grade, 14 June 2018

Zone	Category	Tonnes (Mt)	Zn (%)	Cu (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
Fault Zone	Measured	0.14	1.52	2.74	0.04	16	77	21
	Indicated	1.22	3.32	4.11	0.09	21	96	30
	Inferred	0.20	2.58	3.11	0.07	18	43	23
Série Récurrente	Indicated	0.93	2.43	4.14	0.02	23	50	4
	Inferred	0.03	0.06	1.81	0.00	8	52	0.3
Fault Zone Splay	Inferred	0.21	19.84	4.91	0.01	21	107	93
Total	Measured	0.14	1.52	2.74	0.04	16	77	21
	Indicated	2.15	2.94	4.12	0.06	22	76	19
	Measured and Indicated	2.29	2.85	4.03	0.06	21	76	19
	Inferred	0.44	10.77	3.89	0.04	19	75	55

Contained Metal Quantities								
Zone	Category	Tonnes (Mt)	Zn (Mlb)	Cu (Mlb)	Pb (Mlb)	Ag (Moz)	Co (Mlb)	Ge (Moz)
Fault Zone	Measured	0.14	4.7	8.5	0.1	0.07	0.02	0.09
	Indicated	1.22	89.7	110.8	2.5	0.82	0.26	1.19
	Inferred	0.20	11.1	13.4	0.3	0.12	0.02	0.14
Série Récurrente	Indicated	0.93	49.8	84.6	0.5	0.69	0.10	0.12
	Inferred	0.03	0.04	1.3	0.0	0.01	0.00	0.00
Fault Zone Splay	Inferred	0.21	93.7	23.2	0.1	0.14	0.05	0.64
Total	Measured	0.14	4.7	8.5	0.1	0.07	0.02	0.09
	Indicated	2.15	139.4	195.4	3.0	1.51	0.36	1.31
	Measured and Indicated	2.29	144.2	204.0	3.1	1.58	0.39	1.40
	Inferred	0.44	104.9	37.9	0.4	0.27	0.07	0.78

1. All tabulated data has been rounded and as a result minor computational errors may occur.
2. Mineral Resources are not Mineral Reserves which have no demonstrated economic viability.
3. The Mineral Resource is reported as the total in-situ Mineral Resource and on a 100% project basis, exclusive of Mineral Reserves. Ivanhoe holds an indirect 68% interest in the Project.
4. Metal quantities are reported in multiples of Troy Ounces or Avoirdupois Pounds.
5. The cut-off grade calculation was based on the following assumptions: copper price of $3.00/lb, mining cost of $50/t, processing cost of $10/t, G&A and holding cost of $10/t, 90% copper recovery and 96% payable copper.

Table 14.13 Kipushi – Zinc-Rich Bodies Measured and Indicated Mineral Resource Grade Tonnage Table, 14 June 2018

Cut-off (Zn %)	Tonnes (Mt)	Zn (%)	Zn (Mlb)	Cu (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
5	11.91	35.01	9,193.7	0.81	1.04	23	13	64
7	11.78	35.34	9,176.0	0.80	1.05	23	13	64
10	11.51	35.96	9,125.4	0.78	1.06	23	13	65
12	11.26	36.52	9,063.5	0.76	1.06	23	13	65
15	10.83	37.42	8,937.0	0.73	1.06	23	13	65

1. All tabulated data has been rounded and as a result minor computational errors may occur.
2. Mineral Resources are not Mineral Reserves which have no demonstrated economic viability.
3. The Mineral Resource is reported as the total in-situ Mineral Resource and on a 100% project basis, exclusive of Mineral Reserves. Ivanhoe holds an indirect 68% interest in the Project.
4. Metal quantities are reported in multiples of Troy Ounces or Avoirdupois Pounds.

Table 14.14 Kipushi – Zinc-Rich Bodies Inferred Mineral Resource Grade Tonnage Table, 14 June 2018

Cut-off (Zn %)	Tonnes (Mt)	Zn (%)	Zn (Mlb)	Cu (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
5	1.14	33.77	850.7	1.24	0.24	12	14	62
7	1.14	33.77	850.7	1.24	0.24	12	14	62
10	1.14	33.78	850.6	1.24	0.24	12	14	62
12	1.14	33.91	849.0	1.24	0.24	12	14	61
15	1.11	34.29	842.7	1.21	0.23	12	14	61

1. All tabulated data has been rounded and as a result minor computational errors may occur.
2. Mineral Resources are not Mineral Reserves which have no demonstrated economic viability.
3. The Mineral Resource is reported as the total in-situ Mineral Resource and on a 100% project basis, exclusive of Mineral Reserves. Ivanhoe holds an indirect 68% interest in the Project.
4. Metal quantities are reported in multiples of Troy Ounces or Avoirdupois Pounds.

Table 14.15 Kipushi – Copper-Rich Bodies Indicated Mineral Resource Grade Tonnage Table, 14 June 2018

Cut-off (Cu %)	Tonnes (Mt)	Cu (%)	Cu (Mlb)	Zn (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
1.0	3.72	2.96	242.6	2.10	0.04	17	58	14
1.5	2.29	4.03	204.0	2.85	0.06	21	76	19
2.0	1.55	5.16	175.7	3.59	0.08	26	93	23
2.5	1.20	5.99	158.9	4.08	0.09	30	107	26
3.0	1.00	6.65	146.7	4.43	0.09	33	118	26

1. All tabulated data has been rounded and as a result minor computational errors may occur.
2. Mineral Resources are not Mineral Reserves which have no demonstrated economic viability.

3. The Mineral Resource is reported as the total in-situ Mineral Resource and on a 100% project basis, exclusive of Mineral Reserves. Ivanhoe holds an indirect 68% interest in the Project.
4. Metal quantities are reported in multiples of Troy Ounces or Avoirdupois Pounds.

Table 14.16 Kipushi – Copper-Rich Bodies Inferred Mineral Resource Grade Tonnage Table, 14 June 2018

Cut-off (Cu %)	Tonnes (Mt)	Cu (%)	Cu (Mlb)	Zn (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
1.0	0.55	3.39	40.8	11.90	0.03	17	66	64
1.5	0.44	3.89	37.9	10.77	0.04	19	75	55
2.0	0.35	4.49	34.3	12.21	0.03	20	84	61
2.5	0.29	4.93	31.5	12.14	0.03	21	92	58
3.0	0.24	5.38	28.6	11.18	0.02	22	100	53

1. All tabulated data has been rounded and as a result minor computational errors may occur.
2. Mineral Resources are not Mineral Reserves which have no demonstrated economic viability.
3. The Mineral Resource is reported as the total in-situ Mineral Resource and on a 100% project basis, exclusive of Mineral Reserves. Ivanhoe holds an indirect 68% interest in the Project.
4. Metal quantities are reported in multiples of Troy Ounces or Avoirdupois Pounds.

The Mineral Resource was limited to deeper than approximately 1,150 mL, extensive mining having taken place in the levels above. Below 1,150 mL, some mining has taken place, which has been depleted from the model for reporting of the Mineral Resource. The maximum depth of the Mineral Resource of 1,810 mL is dictated by the location of the diamond drilling data, although sparse drilling completed by KICO below this elevation indicates that the mineralisation has potential to continue at depth. The Mineral Resource occurs close to the DRC-Zambia Border and the Mineral Resource has been constrained to the area considered to be within the DRC.

The Mineral Resource estimate has been completed by Mr J.C. Witley (BSc Hons, MSc (Eng.)) who is a geologist with more than 30 years' experience in base and precious metals exploration and mining as well as Mineral Resource evaluation and reporting. He is a Principal Resource Consultant for The MSA Group (an independent consulting company), which is registered with the South African Council for Natural Scientific Professions (SACNASP) and is a Fellow of the Geological Society of South Africa (GSSA). Mr Witley has the appropriate relevant qualifications and experience to be considered a QP for the style and type of mineralisation and activity being undertaken as defined in National Instrument 43-101 Standards of Disclosure of Mineral Projects.

14.12 Deleterious Elements

The grades of arsenic and cadmium were estimated as shown in Table 14.17.

Table 14.17 Estimated Grades of Arsenic and Cadmium, 14 June 2018

Zone/Class	Arsenic (%)	Cadmium (ppm)
Zinc-Rich Zones (Zn cut-off grade 7%)		
Measured and Indicated	0.16	1,901
Inferred	0.25	1,540
Copper-Rich Zones excluding Fault Zone Splay (Cu cut-off grade 1.5%)		
Measured and Indicated	0.30	202
Inferred	0.12	141
Fault Zone Splay (Cu cut-off-grade 1.5%)		
Inferred	2.94	1,548

14.13 Sulfide Percent Estimates

The sulfide grade of the Kipushi Mineral Resource was assigned to the block model for mining and metallurgical study purposes. The sulfide grades were calculated based on the copper, lead, zinc, and sulfur grade estimates of the block model using the following methodology and assumptions:

- The proportion by weight of each metal in each mineral was calculated:
  - Chalcopyrite 34.643% Cu
  - Galena 86.622% Pb
  - Sphalerite 67.146% Zn
  - Pyrite 46.578% Fe
- The proportion by weight of sulfur in each mineral was calculated as follows:
  - Chalcopyrite 34.915% S
  - Galena 13.378% S
  - Sphalerite 32.854% S
  - Pyrite 53.422% S
- The ratio between sulfur and each metal was calculated:
  - S/Cu in chalcopyrite = 1.008
  - S/Pb in galena = 0.154
  - S/Zn in sphalerite = 0.489
  - S/Fe in pyrite = 1.147
- The total calculated sulfur grade for chalcopyrite, sphalerite, and galena was assigned by dividing the metal grade by the respective sulfur metal ratio for copper, lead, and zinc and added together.

- The total calculated sulfur grade for chalcopyrite, sphalerite, and galena were subtracted from the ordinary kriged sulfur value to derive 'excess sulfur', which was assigned to pyrite.
- The percentage of pyrite was calculated by dividing the 'excess sulfur' grade by the proportion of sulfur in pyrite.
- The percentage of chalcopyrite was calculated by dividing the copper grade by the proportion of copper in chalcopyrite. The percentage of galena and sphalerite were calculated similarly.
- The calculated percent of each of the four sulfides was added together to provide an estimate of total sulfide in each block (CSULFD in the block model). Any value greater than 100% was re-set to 100%.

There are a number of inaccuracies with this method:

- The sulfur/metal ratios assume theoretical values.
- All copper is assumed to be in chalcopyrite, although bornite and other copper minerals exist.
- Sphalerite is assumed to be in a pure form of ZnS. This is never the case and other elements such as iron will occur in the sphalerite.
- Pyrite occurs in the mineralised zones. The calculation assumes any sulfur not assigned to sphalerite, chalcopyrite or galena belongs to pyrite.
- Sulfur is regressed for some holes that did not have sulfur data, which tended to be Gécamines drillholes.
- The sulfur assigned to copper, lead, and zinc can be more than the estimated sulfur grade. The negative 'excess sulfur' grades were retained and used to calculate a pyrite value that was included in the total sulfide calculation.
- It is possible to calculate over 100% sulfides when the zinc grade is very high. This occurred in 0.02% of the sub-blocks and in these cases the estimated sulfide grade was re-set to 100%.

Overall the QP considers that the total sulfide grade assigned to the block model is a reasonable approach in the absence of accurate data in which to estimate the sulfide grade from first principles.

15 MINERAL RESERVE ESTIMATES

Access to the Kipushi Mine will be via vertical shafts and internal decline to the mining zones. Mining is planned to be a combination of Transverse Sublevel Open Stoping and Pillar Retreat mining methods.

The Big Zinc area stopes are planned to be mined as Sublevel Open Stopes to be extracted in a Primary and Secondary sequence, filled with Cemented Rock Fill (CRF). The sill pillars are to be mined on retreat once the stopes below and above have been mined.

The primary and secondary stopes are each 15 m W x 30 m H, with panels of two sublevels separated by a 15 m high sill pillar every 75 m vertically. Stope lengths vary from 5–60 m (maximum). On the sill pillar levels, stopes are 15 m W x 15 m H, with stope lengths varying from 5–60 m.

The stope extraction sequence of primary, secondary then tertiary stopes largely governs which edges of any given stope may be exposed to dilution in the form of over or under break. The quantity and quality of dilution material is determined by understanding the exposure to fill and surrounding rock. The stope dilution factors that have been applied are shown in the below table (Table 15.1).

Table 15.1 Stope Dilution Factors

Primary	0%
Secondary	5%
Tertiary	15%
Southern Zinc	15%

Dilution for the Primary stopes is assumed to be zero as any dilution at this stage will be from surrounding stopes, so any overbreak is assumed to be more of a similar grade to the stope being taken. Subsequent stopes will have dilution from backfill or low-grade material at the mine boundary.

Recovery factors were applied to primary, secondary, and tertiary stopes to account for losses that may occur during the mining process. The factors that have been applied to diluted tonnes are shown in the below table (Table 15.2).

Table 15.2 Stope Recovery Factors

Primary	95%
Secondary	95%
Tertiary	90%
Southern Zinc	95%

The main sources of loss will be from incomplete remote loading, underbreak and overbreak (stope failure). Net Smelter Return (NSR) was used to define the Mineral Reserve cut-offs, and is denominated in $/t. By definition, the break-even cut-off is the point at which the costs are equal to the NSR. An elevated cut-off grade of $135/t NSR was used to define the mining shapes. This elevated cut-off value is in line with the work undertaken during the PFS and previously documented in the NI 43-101 Technical Report Kipushi 2017 Pre-feasibility Study published to System for Electronic Document Analysis and Retrieval (SEDAR) on 25 January 2018.

The break-even cut-off grade has been calculated to be $51.50/t NSR.

A zinc price of $1.10/lb and a treatment charge of $170/dmt concentrate were used in Mineral Reserve estimation, while the economic analysis base case used a zinc price of $1.20/lb and a treatment charge of $190/dmt concentrate.

The Kipushi 2022 Feasibility Study Mineral Reserve has been estimated by Qualified Person, Bernard Peters, Technical Director – Mining, OreWin, using the 2014 CIM Definition Standards. The Mineral Reserve is based on the 14 June 2018 Mineral Resource. The effective date of the Mineral Reserve statement is 14 February 2022. Table 15.3 shows the total Proven and Probable Mineral Reserve of Kipushi.

Table 15.3 Kipushi – Proven and Probable Reserve – Tonnage and Grades

Category	Tonnage (Mt)	Zn (%)	Contained Zn (kt)
Proven	3.33	37.4	1,246
Probable	7.48	29.4	2,199
Total	10.82	31.9	3,445

1. The effective date of the Mineral Reserves is 14 February 2022.
2. Net Smelter Return (NSR) is used to define the Mineral Reserve cut-offs, therefore, cut-off is denominated in $/t. By definition, the cut-off is the point at which the costs are equal to the NSR. An elevated cut-off grade of $135/t NSR was used to define the mining shapes. The marginal cut-off grade has been calculated to be $50/t NSR.
3. The Kipushi 2022 FS Mineral Reserve is based on a zinc price of $1.10/lb Zn and a treatment charge of $170/t concentrate, while the economic analysis to demonstrate the Kipushi 2022 FS Mineral Reserve has used a zinc price of $1.20/lb Zn and a treatment charge of $190/t concentrate.
4. Only Measured Mineral Resources were used to report Proven Mineral Reserves and only Indicated Mineral Resources were used to report Probable Mineral Reserves.
5. Mineral Reserves reported above were not additive to the Mineral Resources and are quoted on a 100% project basis.
6. The Mineral Reserve is based on the 14 June 2018 Mineral Resource.
7. Totals may not match due to rounding.
8. The Proven and Probable Reserve estimate has been reported to conform with the CIM Standards on Mineral Resources (CIM, 2005) of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM).

15.1 Conclusion

Based on the mining production schedule, the criteria applied to the Kipushi Mineral Resource and the cost sensitivity and economic analysis, the Proven and Probable Mineral Reserve has been demonstrated to be viable.

16 MINING METHODS

16.1 Geotechnical

A geotechnical investigation was completed for the Kipushi 2022 FS based on 126 geotechnical borehole logs, 200 m of geotechnical scanline mapping, seven Acoustic televiewer (ATV) logs, and 90 structural borehole logs. Based on the assessments of the data quality it was found that the data available was generally acceptable for use and with minor adjustments was successfully incorporated into the Feasibility Study (FS). Laboratory rock strength testing was also conducted to gain an understanding of the material properties across the project area. Geotechnical parameters based on strategies to manage the potential geotechnical risks have been derived and include backfill strength and support requirements.

16.1.1 Summary of Principal Objectives

The primary aims of the Kipushi feasibility level underground geotechnical investigation and design were as follows:

- To update and improve confidence in the mining geotechnical investigation conducted for the Kipushi PFS, using additional geotechnical and structural data to the level of a FS.
- To optimise the mine design from a geotechnical perspective by conducting numerical analyses based on the FS mine design and data from the mine site.
- To provide geotechnical mine design parameters for the Kipushi Project based on the Big Zinc (BZ) and Southern Zinc (SZ) orebodies to the level of an FS.

16.1.2 Data Collection

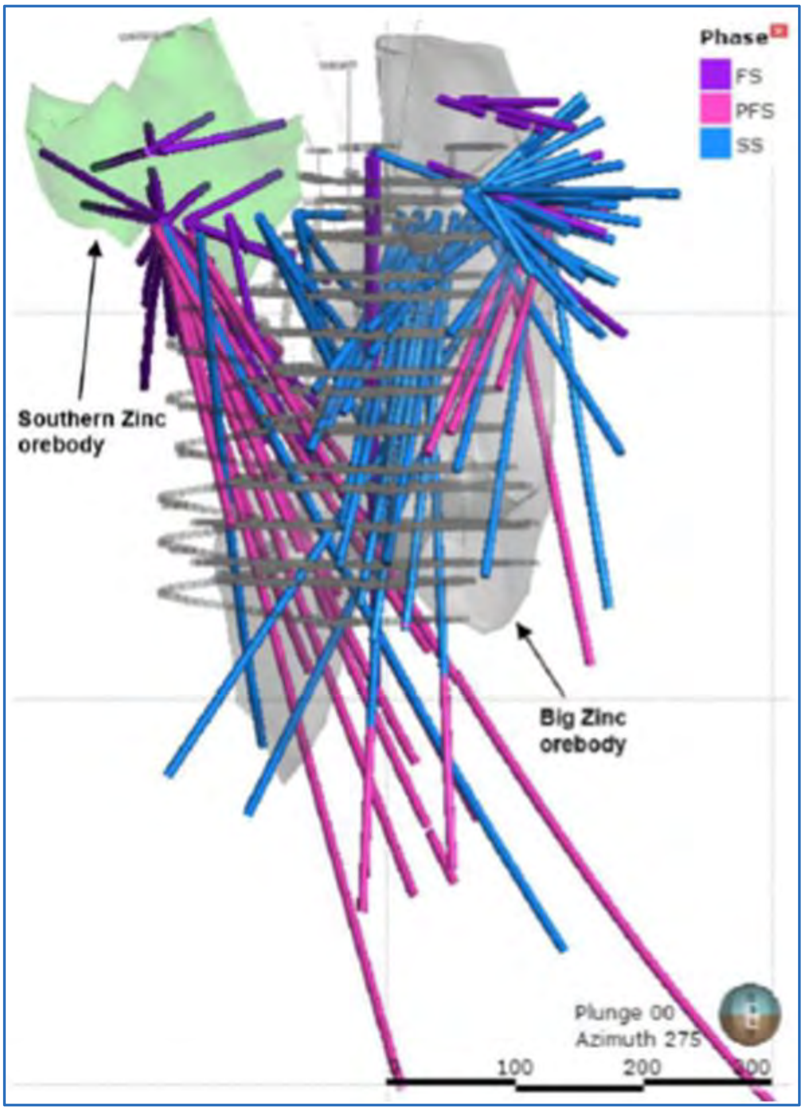
16.1.2.1 Geotechnical Data Collection

Rock mass classification was carried out and a geotechnical block model was created for the Kipushi Project based on geotechnical data available from 126 geotechnical borehole logs (Table 16.1) sourced across the project area (Figure 16.1). It is important to note that the data used includes geotechnical logs from each level of study (i.e. Scoping, Prefeasibility, and Feasibility).

Table 16.1 Kipushi 2022 FS Boreholes

Scoping Study (SS)			Pre-feasibility Study (PFS)				Feasibility Study (FS)		
KPU010	KPU059	KPU075	KPU006	KPU030	KPU047	KPU087	KPU098	KPU113	KPU128
KPU022	KPU060	KPU076	KPU011	KPU031	KPU049	KPU088	KPU099	KPU114	KPU132
KPU024	KPU061	KPU077	KPU012	KPU032	KPU052	KPU089	KPU100	KPU115	KPU134
KPU025	KPU062	KPU078	KPU013	KPU033	KPU053	KPU090	KPU101	KPU116	KPU136
KPU040	KPU063	KPU079	KPU014	KPU034	KPU054	KPU091	KPU102	KPU118	KPU137
KPU042	KPU064	KPU080	KPU015	KPU035	KPU073	KPU092	KPU103	KPU119	KPU138
KPU046	KPU065	–	KPU016	KPU036	KPU074	KPU093	KPU104	KPU120	KPU139
KPU048	KPU066	–	KPU017	KPU037	KPU028	KPU093w1	KPU105	KPU121	KPU143
KPU050	KPU067	–	KPU018	KPU038	KPU081	KPU094	KPU106	KPU122	KPU151
KPU051	KPU068	–	KPU019	KPU039	KPU082	KPU095	KPU107	KPU123	KPU152
KPU055	KPU069	–	KPU021	KPU041	KPU083	KPU096	KPU108	KPU124	–
KPU056	KPU070	–	KPU023	KPU043	KPU084	KPU097	KPU109	KPU125	–
KPU057	KPU071	–	KPU026	KPU044	KPU085	–	KPU111	KPU126	–
KPU058	KPU072	–	KPU029	KPU045	KPU086	–	KPU112	KPU127	–

Figure 16.1 Boreholes per Level of Study at Kipushi – Looking North-West



SRK, 2019

16.1.2.2 Structural Data Collection

A structural analysis was conducted for the Kipushi Project area based on the following data:

- Underground geotechnical mapping conducted by SRK Consulting (South Africa) (Pty) Ltd (SRK) on 1,220 mL in October 2016 (Table 16.2, Table 16.3, and Figure 16.2). Note that joint data was also briefly collected on 1,270 mL.
- Acoustic televiewer (ATV) data collected by Gap Geophysics from seven boreholes which intersect the BZ orebody, provided to SRK by senior geologist Tim Dunnett from Kipushi Mine in December 2017 (Table 16.4).

- Detailed geotechnical structural borehole logging data and geological structural logs from a total of 90 boreholes, which have been collected by Kipushi representatives since the Kipushi scoping study to date (Table 16.5 and Figure 16.3). Please note that boreholes logged geotechnically are logged in detail, where every discontinuity was logged, whilst geologically logged holes only identify major structures.

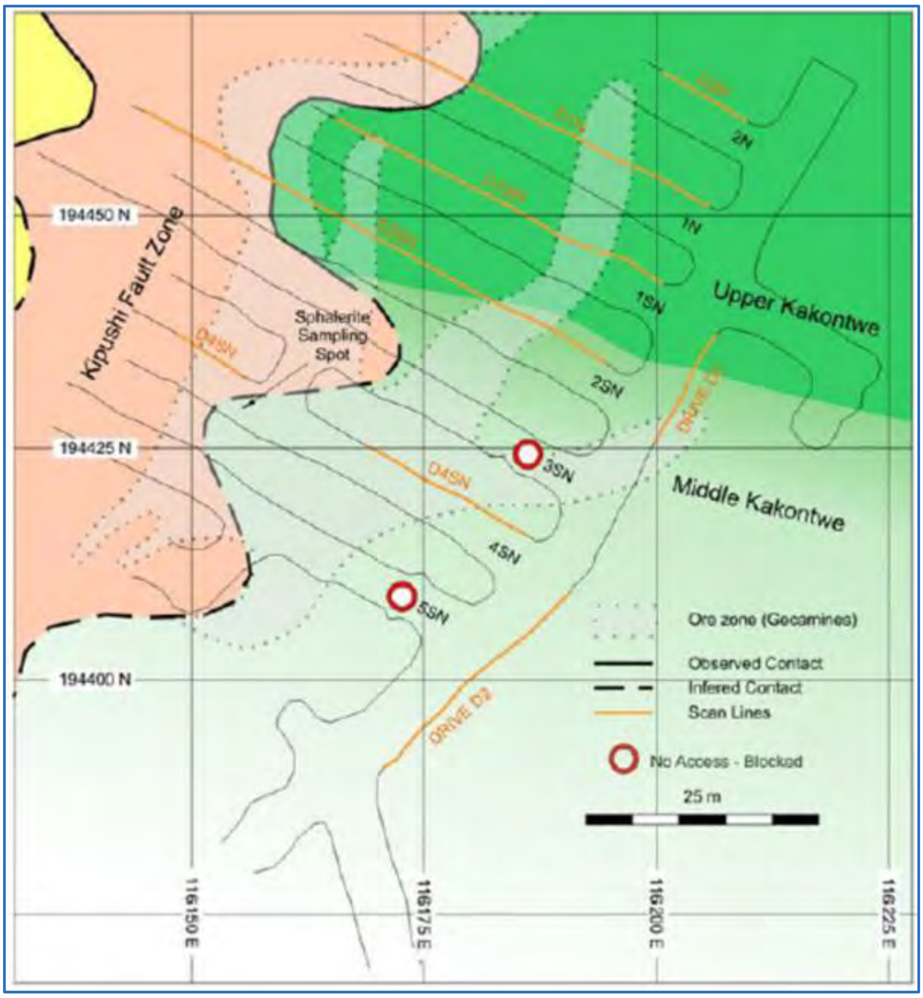
Table 16.2 Mapping Conducted on 1,220 mL

Location	Formation	Rock Unit	Total Length Mapped (m)
2N	Dolomite	Upper Kakontwe	11.25
1N	Dolomite	Upper Kakontwe	35.55
1SN	Dolomite	Upper Kakontwe	39.95
2SN	Dolomite Sphalerite Siltstone	Upper Kakontwe BZ Grand Lambeau	43.90 1.70 12.7
3SN	No Access		
4SN	Dolomite Sphalerite	Middle Kakontwe BZ	29.15 12.10
5SN	No Access		
Drive (D2)	Dolomite	Middle Kakontwe	35.35
Drive (D3)	Dolomite	Upper Kakontwe	9.60

Table 16.3 Summary of Mapping Conducted

Rock Unit	Formation	Location	Total Length Mapped (m)	No. of Joints Identified
Dolomite	Upper Kakontwe	1N, 2N, 1SN, 2SN, D3	139.25	467
Dolomite	Middle Kakontwe	4SN, D2	64.50	158
Sphalerite	Big Zinc Orebody	2SN, 4SN	13.80	35
Siltstone	Grand Lambeau	2SN, 1,270 mL	12.70	72

Figure 16.2 Mapping Conducted on 1,220 mL



SRK, 2019

Table 16.4 Kipushi – ATV Boreholes

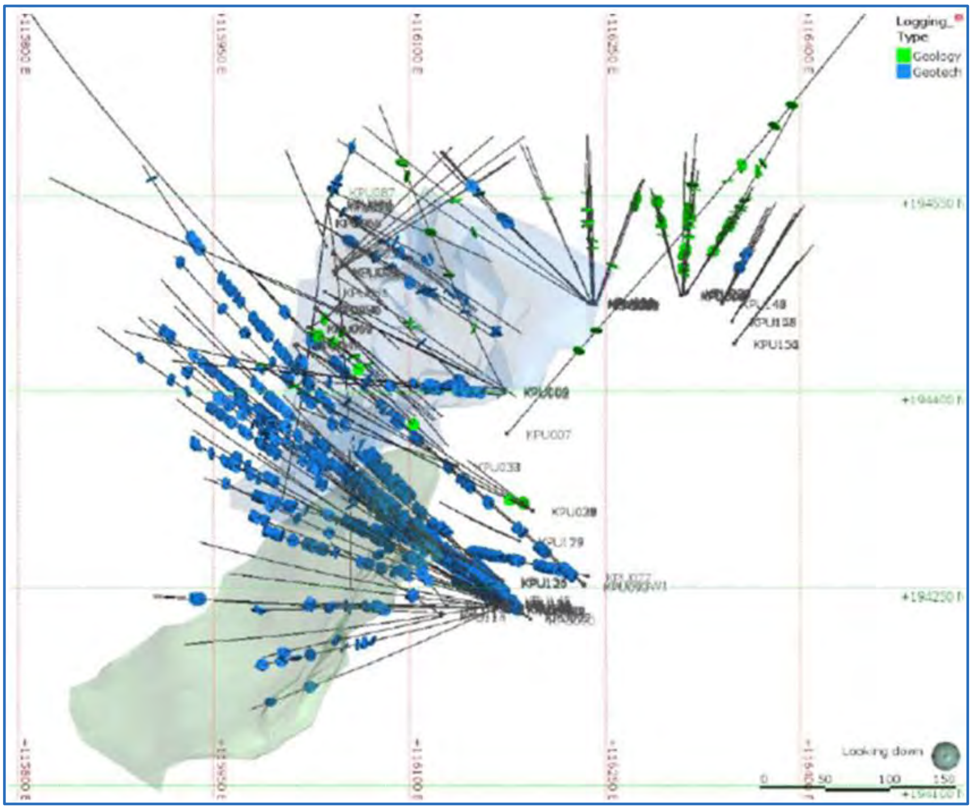
Borehole ID	X (m)	Y (m)	Z (m)	Dip (Degrees)	Azimuth (Degrees)
KPU002	116,173.5	194,400.1	–1,221.51	57	295
KPU037	116,135.9	194,343.2	–1,270.58	45	291
KPU052	116,243.4	194,466.2	–1,257.13	44	21
KPU053	116,242.3	194,466.7	–1,257.12	45	356
MPU064	116,023.4	194,449.9	–1,268.90	67	128
KPU075	116,037.4	194,541.5	–1,267.21	56	177
KPU086	116,038.2	194,543.9	–1,267.67	51	126

Table 16.5 Kipushi – Structural Boreholes

*KPU004	*KPU026	*KPU057	KPU073	KPU086	KPU106	KPU125
*KPU005	*KPU028	*KPU058	KPU074	KPU087	KPU107	KPU126
*KPU006	*KPU030	*KPU059	KPU075	KPU088	KPU108	KPU127
*KPU007	*KPU032	*KPU061	KPU076	KPU089	KPU109	KPU128
*KPU008	*KPU036	*KPU062	KPU077	KPU090	KPU111	KPU132
*KPU009	*KPU040	KPU065	KPU078	KPU091	KPU112	KPU134
*KPU010	*KPU042	KPU066	KPU079	KPU093	KPU113	KPU136
*KPU011	*KPU044	KPU067	KPU080	KPU093W1	KPU114	KPU137
*KPU014	*KPU051	KPU068	KPU081	KPU094	KPU115	KPU138
*KPU016	*KPU052	KPU069	KPU082	KPU097	KPU116	KPU139
*KPU018	*KPU053	KPU070	KPU083	KPU099	KPU118	KPU143
*KPU022	*KPU054	KPU071	KPU084	KPU102	KPU122	KPU151
*KPU023	*KPU056	KPU072	KPU085	KPU104	KPU124	–

*Geological logs

Figure 16.3 Extent of Structural Borehole Logging Data



SRK, 2019

A summary of the structural data used in the Kipushi PFS compared with the data utilised in the Kipushi FS is presented in Table 16.6.

Table 16.6 Summary of Data Collected

Data Available	Source/Total No. of Holes	
	PFS only	FS (includes PFS data)
Underground Geotechnical Mapping	1,220 mL	1,220 mL
Structural Logs	60	90
ATV logs	0	7

16.1.2.3 Rock Sampling for Laboratory Testing

During the Kipushi SS and PFS studies, rock strength tests were carried out to gain an impression of the intact rock strength properties of the major lithological units present within and in the vicinity of the BZ orebody. Further sampling and testing was then carried out for the FS (Table 16.7), with the aim to improve confidence the strength properties determined for the BZ and surrounding rock and to investigate the strength properties within the SZ orebody.

To determine the strengths of the rock units, a laboratory testing programme was compiled which comprised the following geomechanical tests:

- Uniaxial Compressive Strength with Young's Modulus and Poisson's Ratio (i.e. UCM).
- Uniaxial Compressive Strength (UCS).
- Uniaxial Indirect Tensile (UTB) Strength (Brazilian method).
- Base friction angle tests (BFA).

All testing was conducted by Rocklab, which is located in Pretoria, South Africa.

Table 16.7 Summary of Tests Conducted

Study	Lithology	Number of Samples					Subtotal
		UCS	UCM	TCS	UTB	BFA	
Scoping	Dolomite (SDO)	8	–	–	–	–	30
	Shale (SSH)	5	–	–	–	–	
	Siltstone (SSL)	5	–	–	–	–	
	Sandstone (SST)	5	–	–	–	–	
	Sphalerite (SPH)	4	–	–	–	–	
	Mixed Sulfide Minerals (MSM)	3	–	–	–	–	
PFS	Dolomite	6	20	33	38	16	173
	Siltstone	–	4	9	6	2	
	Sandstone	–	2	3	8	3	
	Sphalerite	–	5	9	5	4	
FS	Dolomite	–	7	26	23	–	101
	Siltstone	–	6	3	–	–	
	Sphalerite	–	–	17	–	–	
	Mixed Sulfide Minerals	–	2	3	–	–	
	Chalcopyrite (CPY)	–	7	–	–	–	
	Pyrite (PYR)	–	4	3	–	–	
Subtotal		35	57	106	80	25	
Total number of tests		304					

The majority of the rock samples selected for testing were obtained from borehole core and comprised of the following rock units:

- Kakontwe dolomite (present in the SZ footwall and in the BZ hanging wall and footwall, and sometimes exists within the orebody).
- Siltstones, shales, and sandstones of the Grand Lambeau and the Kipushi fault zone (present in the BZ and SZ hanging walls).
- Sphalerite (dominant lithology in the orebody).
- Pyrite, chalcopyrite, and mixed sulfide minerals (minor mineralised rock units, which are not often present but do exist in the project area).

Rock samples were also chosen based on the following:

- Availability and accessibility.
- Absence of discontinuities – only core which appeared free from discontinuities were selected.
- Location – hanging wall and footwall samples were selected as close to the orebodies as possible.

It is noted that as it was difficult to source the required number of samples for testing of the sphalerite from borehole core during the PFS, therefore, block samples were also selected for this unit on 1,220 mL during the sampling exercise conducted in October 2016.

Samples collected in November 2017 from the various orebodies and the surrounding rock to supplement samples collected during the SS and PFS are summarised in Table 16.8.

Table 16.8 Samples Collected for Inclusion into FS Laboratory Testing Programme

Location	Rock Types Tested	UCM	TCS	BTS	Total No. of Samples
Big Zinc	SPH, CPY, MSM, PYR, SDO	5	15	–	20
Southern Zinc	SPH, PYR, SDO	3	9	–	12
Série Récurrente	CPY	3	–	–	3
Copper Nord Riche	CPY	3	–	–	3
Kakontwe SDO	SDO	6	25	23	54
GLB/KFZ	SSL	6	3	–	9
Total		101			

16.1.2.4 Quality Assurance

Geotechnical logging and laboratory sampling were reviewed as a quality control measure during each level of study for the Kipushi Project including the FS. A visit to the Kipushi Mine was conducted by SRK in November 2017 to assess the geotechnical data collected based on the 2017 drilling campaign and to inspect rock conditions prior to the commencement of the Kipushi FS.

Geotechnical and geotechnical structural logging based on the 2017 drilling campaign have been conducted by contracted geologists under the supervision of Mr Tim Dunnett (senior geologist) of Ivanhoe mines. Whilst on site eight borehole logs were assessed. Boreholes chosen for the assessment of the logging are presented in Table 16.9.

Table 16.9 Borehole Logging Reviewed – QA/QC

Borehole ID	Logged by	Type of Logging Reviewed
KPU102	Yannick Kasongo / Guedally Mahako	Geotechnical
KPU112	Fanfan Ilunga / Yves Ramazani	Geotechnical
KPU112	Fanfan Ilunga / Yves Ramazani	Structural
KPU139	Malick Ndula / Joachim Muteba	Geotechnical
KPU132	Christopher Sadini / Trang Banza	Geotechnical
KPU132	Christopher Sadini / Trang Banza	Structural
KPU128	Joachim Muteba / Malick Ndula	Geotechnical
KPU128	Joachim Muteba / Malick Ndula	Structural

From the assessment of the logging on site and from further evaluation in the Johannesburg office with the use of core photographs the following was observed:

- An entirely new team of personnel from Kipushi carried out the geotechnical logging of the FS holes. As this team had little experience in geotechnical logging, a conservative approach was employed.
- The geotechnical logging is based on Laubscher's MRMR (Mining Rock Mass Rating) system. This method allows for the allocation of joints into three categories (0–30°, 30–60° and 60–90°). This system is favoured as it allows for the determination of both Laubscher MRMR values and Barton Q values, which is required for the Kipushi FS.
- Geotechnical intervals are well defined and are based on changes in geology and/or significant changes in the degree of fracturing and the general condition of the core.
- Joints are clearly marked on the core and are demarcated based on their respective joint category (0–30°, 30–60° or 60–90°, relative to the core axis).
- Mechanical breaks are not always marked on the core. Mechanical breaks should be marked to avoid confusion between these breaks and natural occurring fractures.
- Joint roughness and joint infill are being logged to an acceptable standard, however, in certain instances haematite staining is recorded as thin infill. This is considered conservative (This was generally logged as staining by the SS and PFS team). This results in higher Ja values and thus lower Q values in the FS logs compared with the SS and PFS logs.
- There are sometimes discrepancies in the beta angles recorded. This was further investigated and rectified by Kipushi representatives.

Generally, the logging completed for the 2017 boreholes is conservative. This resulted in lower Q values compared with the logging conducted by a separate (more experienced team) during the SS and PFS phase. While this is the case, the logging is of an acceptable standard and with minor amendments, was utilised successfully in the Kipushi FS.

16.1.2.5 Summary of Data Collected

Overall 126 geotechnical borehole logs, 200 m of geotechnical scanline mapping, seven ATV logs, 90 structural borehole logs and 304 laboratory strength tests were utilised in the FS. Based on the assessments of the data quality it was found that the data available was generally acceptable for use and with minor adjustments was successfully incorporated into the FS.

16.1.3 Rock Properties

16.1.3.1 Results from Testing

A summary of the laboratory tests results for all lithologies tested are presented in Table 16.10 and the description of rock units is presented in Section 16.1.2.3.

Table 16.10 Laboratory Testing Results

Material Property	Rock Unit	SDO	SPH	SSL	SSH	SST	MSM	CPY
Density (kg/m^3)	Number of Tests	161	40	33	5	18	8	7
	Minimum	2.7	3.62	2.7	2.76	2.7	3.12	3.8
	Mean	2.85	4	2.8	2.82	2.74	4.19	3.94
	Maximum	3.03	4.38	2.86	2.86	2.83	4.27	4.02
	Standard Deviation	0.04	0.18	0.03	0.04	0.03	0.42	0.08
UCS (Mpa)	Number of Tests	41	9	15	5	7	5	7
	Number of successful tests	37	5	4	2	3	1	1
	Minimum	149	123	175	215	192	326	212
	Mean	290	220	236	247	253	326	212
	Maximum	391	318	315	279	307	326	212
	Standard Deviation	49	70	64	42	58	–	–
UTB (Mpa)	Number of Tests	61	5	6	0	8	0	0
	Minimum	7	5	16	–	13	–	–
	Mean	13	7	17	–	17	–	–
	Maximum	19	10	18	–	20	–	–
	Standard Deviation	3	2	1	–	2	–	–
BFA (0)	Number of Tests	16	4	2	0	3	0	0
	Minimum	23	19	30	–	34	–	–
	Mean	34	26	34	–	35	–	–
	Maximum	42	29	38	–	36	–	–
	Standard Deviation	6	5	6	–	1	–	–

Note that statistics in Table 16.10 are based only on successful tests (samples that failed along discontinuities are not included). The number of rock strength test carried out for the Dolomite (SDO) is 41 with 37 successful tests. The SDO footwall rock strengths are very high (mean rock strength of 307 Mpa), however, there is high confidence in the test results. The SDO represents 75% of the footwall lithologies (Section 16.1.2.3) based on the information from the geotechnical bore holes. The bore holes represent the immediate footwall lithologies where most of the future development is planned. It is therefore, recommended that where large excavation or raise boring is planned in the footwall that a pilot hole be drilled for rock property testing and geotechnical logging (Section 16.1.5).

Material properties σ_{ci} and mi were obtained from the available UCS, UCM, UTB, and TCS test results, whereby, a Hoek-Brown failure envelope was fit to the data for the major rock units.

The hanging wall of the mineralised zone is represented by Grand Lambeau/ Kipushi Fault Zone (GLB/KFZ) and the footwall by the Kakontwe Formation. The BZ and SZ orebodies comprise the sphalerite (SPH) rock material (mineralised zone). A summary of the laboratory tests results for SPH are presented in Table 16.11.

Table 16.11 Laboratory Testing Results for SPH

Material Property	Rock Unit	SPH – BZ	SPH – SZ
UCS (Mpa)	Total number of Tests	9	0
	Number of successful tests	5	–
	Minimum	123	–
	Mean	220	–
	Maximum	318	–
	Standard Deviation	70	–
UTB (Mpa)	Number of Tests	5	0
	Minimum	5	–
	Mean	7	–
	Maximum	10	–
	Standard Deviation	2	–
Total Number of Successful Tests		18	4

A summary of the Hoek-Brown properties determined is presented in Table 16.12. The variability of the rock strengths is high, therefore, the mean rock strengths were used, which is representative of the rock mass. The mean rock strength will be used for the stope design and assessment of development stability.

Table 16.12 Hoek-Brown Properties

Hoek-Brown	Rock Unit	GLB/KFZ	BZ/SZ	SDO
	Total number of Tests	56	40	161
	No. of tests which failed along discontinuities	32	17	22
	σ_{ci} – 1 std deviation	124	146	251
	Mean σ_{ci}	188	198	307
	σ_{ci} + 1 std deviation	253	250	364
	Standard Deviation	64	52	64
	Mi	11	21	23

Elastic properties for the major units are presented in Table 16.13. Note that the elastic properties of the intact rocks are represented by Young's Modulus and Poisson's Ratio, which were derived from UCM tests where the measurement of deformation during the tests were recorded with the use of strain gauges.

Table 16.13 Elastic Properties

Material Property	Rock Unit	GLB/KFZ	BZ/SZ	SDO
Young's Modulus (Gpa)	Total number of Tests	12	5	27
	Minimum	47	54	49
	Mean	71	64	90
	Maximum	96	75	117
	Standard Deviation	15	8	21
Poisson's Ratio	Total number of Tests	12	5	27
	Minimum	0.2	0.22	0.1
	Mean	0.25	0.26	0.26
	Maximum	0.3	0.3	0.35
	Standard Deviation	0.03	0.03	0.05

16.1.3.2 Summary of Results and Recommendations

Based on the analysis of the laboratory results the following may be concluded:

- While great care was taken to ensure that samples chosen were free from apparent discontinuities, it was observed that many samples tested in the GLB/KFZ failed along pre-existing discontinuities. This is likely due to the laminated nature of the siltstone and shale in this stratigraphy. While these tests failed along discontinuities, it was found that the results fit reasonably well within the Hoek-Brown curves determined.

- Where the hanging wall comprises of the GLB/KFZ, an intact rock strength of approximately 188 Mpa should be expected (Table 16.12).
- Where the orebody is intersected, an intact rock strength of approximately 198 Mpa should be anticipated (Table 16.12). Note that higher strengths are likely where minor units (e.g. PYR, MSM, CPY) and the Kakontwe SDO is present (Table 16.10).
- Overall, the strength of the rock units within the GLB/KFZ and the BZ may be defined as very strong rock (Table 16.14).
- Where the hanging wall comprises of the Kakontwe SDO (which is sometimes the case in the deeper levels for the BZ), very high strengths (approximately 307 Mpa) should be anticipated. This high strength should also be expected in the footwall as this mining unit consists predominantly of Kakontwe SDO.
- The Kakontwe SDO is the strongest unit in the project area, which may be classified as extremely strong rock (Table 16.14).
- All strengths determined for the major rock units in the FS lie within the same strength classification bracket compared with the strengths determined during the PFS (Table 16.15). Overall, the GLB/KFZ FS strengths are somewhat lower compared with the PFS results. This is because all test results were incorporated at the FS level of the study while only 'successful' tests were included at PFS level.

Table 16.14 Rock Strength Classification

Strength (Mpa)	Strength Classification
<1	Extremely weak rock
1 – 5	Very weak rock
5 – 25	Weak rock
25 – 50	Medium strong rock
50 – 100	Strong rock
100 – 250	Very strong rock
>250	Extremely strong rock

ISRM, 1981

Table 16.15 Comparison of FS with PFS Results

Rock Unit	**Mean σ_{ci}** (Mpa)		Strength Classification
	PFS	FS	
GLB/KFZ	223	188	Very strong rock
BZ	208	198	Very strong rock
Kakontwe SDO	275	307	Extremely strong rock

The results from testing have been utilised for rock mass characterisation, the empirical stope design and for numerical modelling purposes.

16.1.4 Regional Stress Field and Seismic Hazard

16.1.4.1 Regional Stress Field

As there is no stress data currently available for the Kipushi Mine, the regional stress field is estimated from the World Stress Map (Heidbach et al., 2016) and the Seismotectonic Map of Africa (Meghraoui et al., 2016). These maps have been compiled from the orientations of earthquake focal mechanisms, borehole breakout, drilling-inducing fractures, hydraulically induced fractures, and structural mapping data.

According to both maps, Kipushi is located in an extensional regime where SV > Shmax > Shmin and the maximum horizontal stress is orientated approximately north–north-east. At this stage of the project, the anisotropic horizontal field stress have not been included in the elastic modelling (Section 16.1.7), but the natural horizontal stress have been estimated on the basis of stress damage or lack thereof, observed underground (SH = 0.8 SV). It is recommended that field stress measurements be carried out during the execution stage of the project. This will help to confirm the onset of stress damage and anticipated rock behaviour in future.

16.1.4.2 Seismic Hazard

The Kipushi Mine is situated in a moderately seismically active area of the Congo Craton in a seismogenic zone referred to as the Upemba-Moero Rift zone. Mavonga and Durrheim (2009) performed a probabilistic seismic hazard assessment of the DRC and surrounding areas. For the Upemba-Moero Rift seismogenic zone in which Kipushi is situated, peak ground accelerations (PGA) are estimated from 0.05–0.09 g for a 10% chance of exceedance in 50 years. PGA exceeding 0.05 g is the threshold value of engineering interest and should be taken into consideration in the design of surface structures.

The Kipushi rocks are very strong and brittle. At this stage, the stress damage observed underground is limited, but as the mining depth increases, the stress will increase, and the walls of excavations will experience stress damage. Due to the high strength and brittle nature of the rock, strainbursts and rockbursts can be anticipated. Numerical modelling was carried out to estimate the depth of failure in the walls of excavations, in order to determine the support requirements (Section 16.1.7). Dynamic support has been recommended when the depth of failure exceeds 0.5 m.

16.1.5 Geotechnical Model

16.1.5.1 Rock Mass Classification

To classify the quality of the rock mass in the vicinity of the BZ and SZ orebodies, use was made of three rock mass classification systems, viz. Barton et al's (1974) Norwegian Geotechnical Institute's Q-System, modified Q' (Barton and Grimstad, 1974 and Barton 2002), Laubscher's (1990) Mining Rock Mass Rating Classification System, and Hoek's (2003) quantification of the Geological Strength Index (GSI) system.

For the application of the statistical methods, a weighted averaging method known as compositing was applied to the data to produce geotechnical intervals of equal lengths, allowing for statistical analysis. This operation was performed using LEAPFROG and GEMCOM computer software. An interval (compositing) length of 3 m was chosen for the data as this was the typical core run length and was applied throughout the orebody. Compositing was extended 5 m into the hanging wall and 50 m into the footwall. The 5 m compositing into the hanging wall takes into consideration the zone that is likely to influence the stability of the stopes and the 50 m into the footwall, is the zone where access development will be located. The compositing was applied to each geotechnical interval for every available borehole. Rock mass classification results are presented from Table 16.16 to Table 16.20.

Table 16.16 Comparison of RMR L90 Results for BZ and SZ

Orebody	Mining Position	Number of Boreholes	Minimum	Mean	Maximum	Standard Deviation
BZ	5 m HW	56	46	82	100	17
	OB	54	26	83	100	17
	50 m FW	53	46	79	100	14
SZ	5 m HW	18	47	61	86	10
	OB	15	32	76	100	18
	50 m FW	18	52	66	78	7

Table 16.17 Q' Results for SZ and BZ

Orebody	Mining Position	Number of Boreholes	Minimum	Median	Maximum	20 percentile	80 percentile
BZ	5 m HW	56	6	110	708	20	708
	OB	54	0.3	286	711	17	710
	50 m FW	53	4	71	708	34	411
SZ	5 m HW	17	1	12	676	5	36
	OB	15	2	43	710	13	224
	50 m FW	18	4	8	71	5	17

Table 16.18 Q' Results for BZ and SZ

Orebody	Mining Position	Number of Boreholes	Minimum	Median	Maximum	20 percentile	80 percentile
BZ	5 m HW	56	7	112	708	29	708
	OB	54	2	453	711	28	711
	50 m FW	53	7	71	708	36	708
SZ	5 m HW	17	4	19	676	11	676
	OB	15	4	87	710	30	710
	50 m FW	18	6	19	85	11	85

Table 16.19 RQD Results for BZ and SZ

Orebody	Mining Position	Number of Boreholes	Minimum	Median	Maximum	20 percentile	80 percentile
BZ	5 m HW	56	67	98	100	93	100
	OB	54	10	98	100	90	100
	50 m FW	53	56	96	100	95	99
SZ	5 m HW	17	12	79	97	66	93
	OB	15	50	94	100	72	99
	50 m FW	18	75	95	97	87	97

Table 16.20 Comparison of GSI Results for BZ and SZ

Orebody	Mining Position	Number of Boreholes	Minimum	Mean	Maximum	Standard Deviation
BZ	5 m HW	56	49	81	94	12
	OB	54	23	82	94	14
	50 m FW	42	52	79	94	11
SZ	5 m HW	18	23	64	91	17
	OB	15	42	76	94	13
	50 m FW	18	54	71	81	7

16.1.5.2 Underground Conditions

Underground visits were conducted for the SS in May 2014 and for the PFS in October 2016. Further underground visits were conducted for the FS in November 2017 and August 2018. The purpose of the underground visits was to assess rock conditions on the results of existing excavations, verify the rock mass classification, results determined from core logging, and to inspect areas not covered by the geotechnical logging.

During the visits, the following areas were inspected:

- Workshops and crusher chamber 1,132 mL and 1,150 mL,
- Decline,
- Production levels 1,182 mL, 1,195 mL, and 1,220 mL, and
- Shafts and winzes P2, P2B, and P19.

The mine was flooded in 1993 and then dewatered. Despite this, most of the areas are in reasonably good condition.

Overall, the ground conditions observed underground are good and much of the existing infrastructure and access development is safe without any support. The observed underground conditions correlate to the quality of ground obtained from the rock mass classification. There are areas where poor ground conditions occur, and these areas should be identified and inspected to confirm that the current support is appropriate and in reasonable condition. New support should be installed where required.

Given that the major infrastructure near the shafts is located in laminated rock, where fair to poor ground conditions occur, it is recommended that, these areas are examined regularly and re-supported as necessary, even though the area has generally been supported appropriately and may have been subsequently rehabilitated.

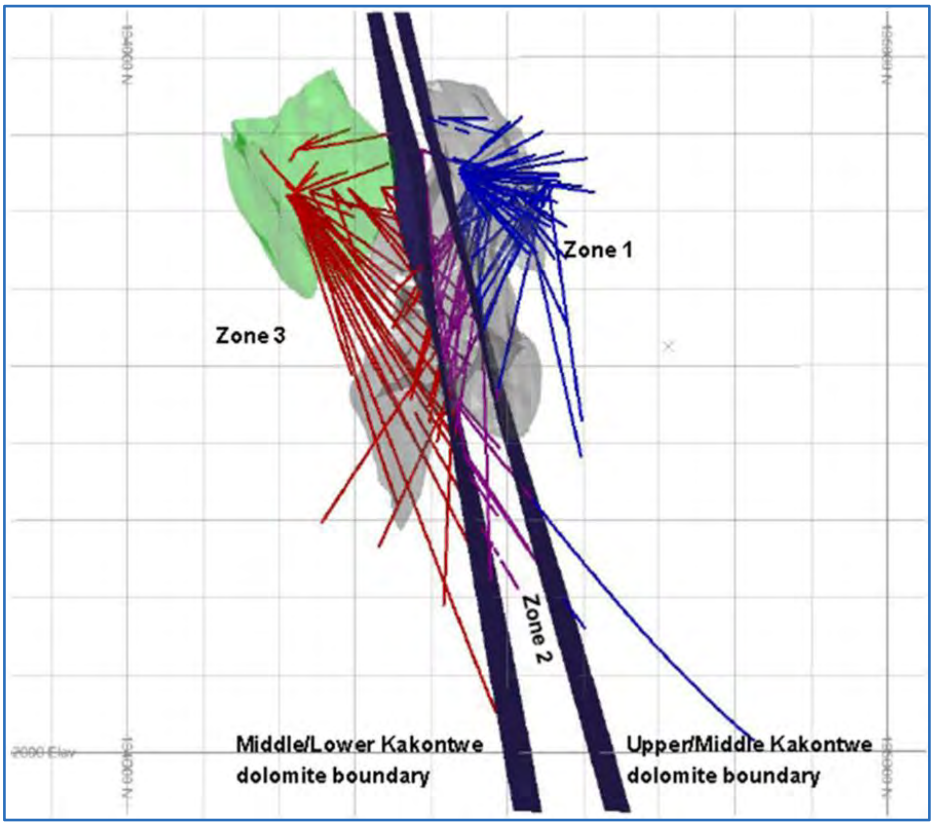
16.1.5.3 Geotechnical Block Model

A geotechnical block model was created for the Kipushi Mine project based on geotechnical data available from 126 geotechnical borehole logs from boreholes located across the project area. This was done to provide a 3D visual representation of the rock mass conditions across the project area and to allow for the identification of potential poor ground conditions and provide the opportunity to plan for these conditions accordingly.

Geotechnical Domains

On analysis of the rock quality across the project area, it was observed that overall the rock mass quality is highest in the middle of the project area (where the majority of the BZ is located) compared with the north and south of the project area. It was decided to separate the data into three zones. As the poorer quality rock in the north and south is likely due to the more fractured nature of the upper and lower Kakontwe dolomite, these boundaries were utilised as a guideline to separate the zones (Figure 16.4).

Figure 16.4 Geotechnical Zones – Looking North-West



SRK, 2019

Geotechnical Model Creation

Datamine Studio RM computer software was used to generate the Kipushi geotechnical block model.

For the creation of the model, variograms were required and thus created in three orthogonal directions to gain an impression of the spatial continuity of the data across the project area. It was observed in the experimental semi-variograms, that the data has the longest range of continuity in the vertical direction. Using these results, a variogram model was created for Kipushi.

Two methods were employed for the creation of the block model:

- Nearest Neighbour, and
- Ordinary Kriging.

To honour the data within the boreholes, the nearest neighbour method was applied. This method does not involve weighting sample values. Instead, each cell is assigned the value of the 'nearest' sample, where 'nearest' is defined as a transformed or anisotropic distance which takes account of any anisotropy in the spatial distribution of the log Q values.

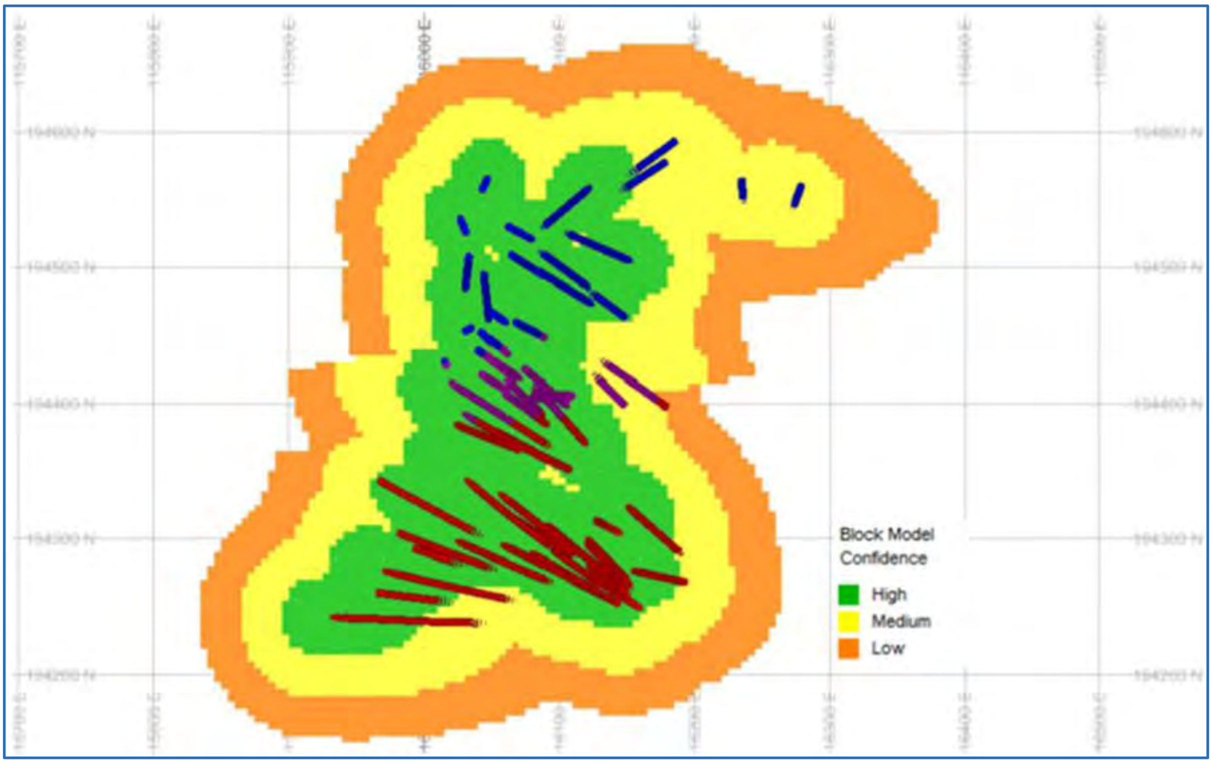
Ordinary kriging was applied to the Kipushi data using a three-search pass strategy, where the distance from the data was incrementally increased for each search pass (Table 16.21). This was done to increase the smoothing of the block model as the distance from the data increased, while locally honouring the nearby data. The ranges chosen for each search pass were based on the variogram results. For each search pass, a minimum and maximum number of samples to be utilised was defined. Note that where more than the maximum number of samples within search volume exist, the nearest samples are selected.

Table 16.21 Search Pass Parameters

Search Pass	Range (m)	Minimum Number of Samples	Maximum Number of Samples
1	30	6	10
2	60	6	12
3	90	6	20

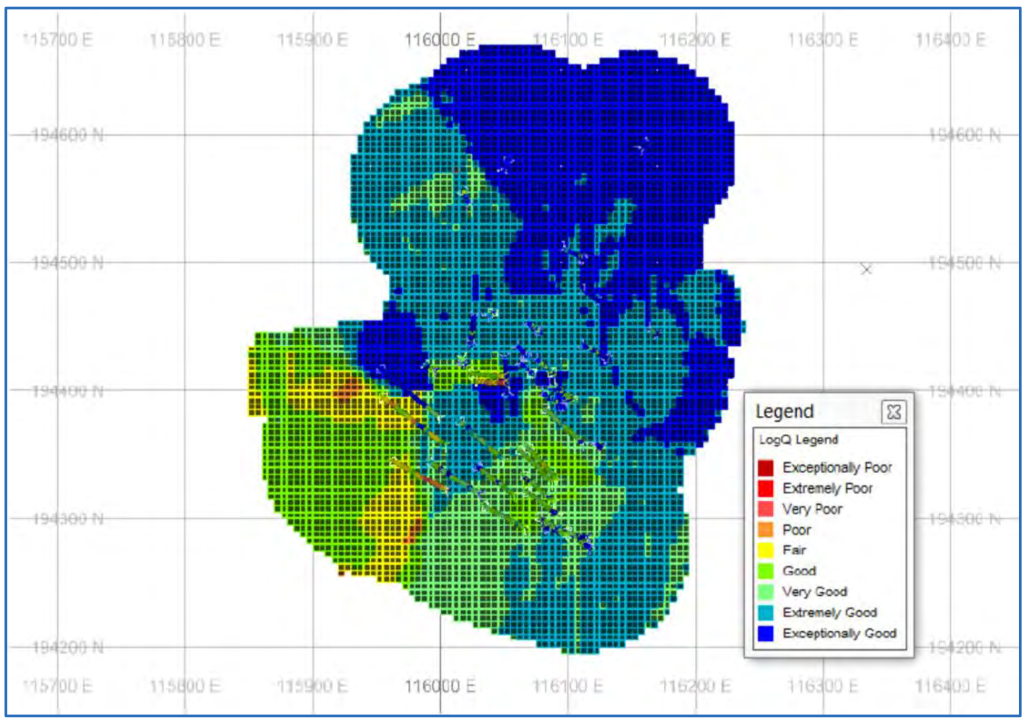
Figure 16.5 illustrates the confidence in the block model, which decreases as the distance from the boreholes increase. As there is no data available in the far east of the project area note that this was not modelled. Sections through the Kipushi block model are presented in Figure 16.6.

Figure 16.5 Block Model Confidence (Plan View at 1,390 mL)



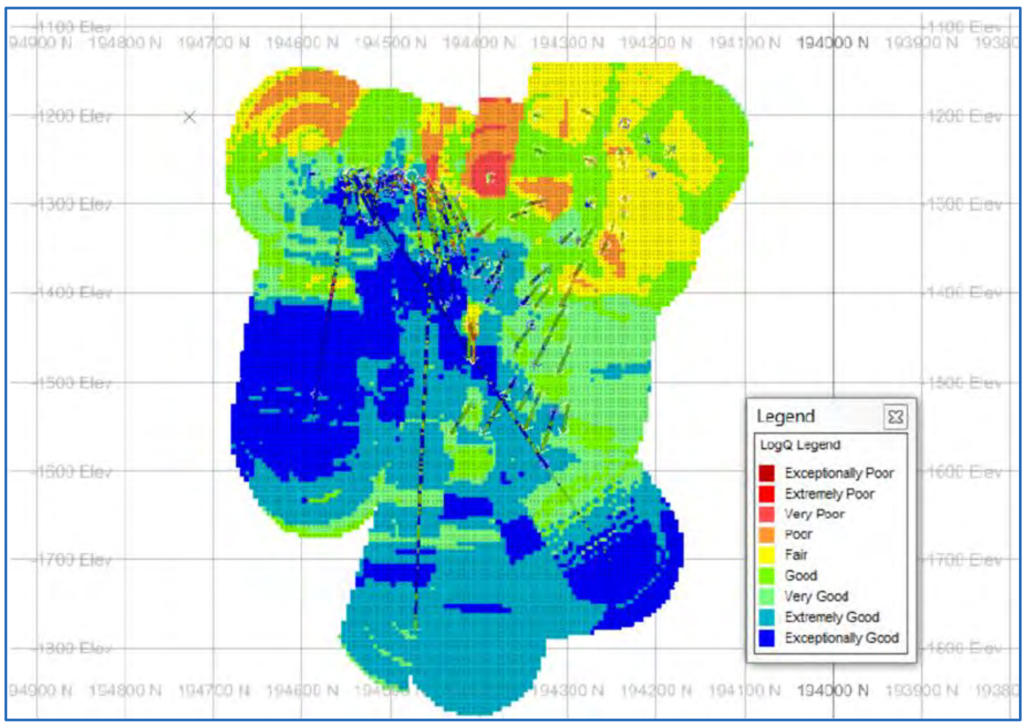
SRK, 2019

Figure 16.6 RMR from Q



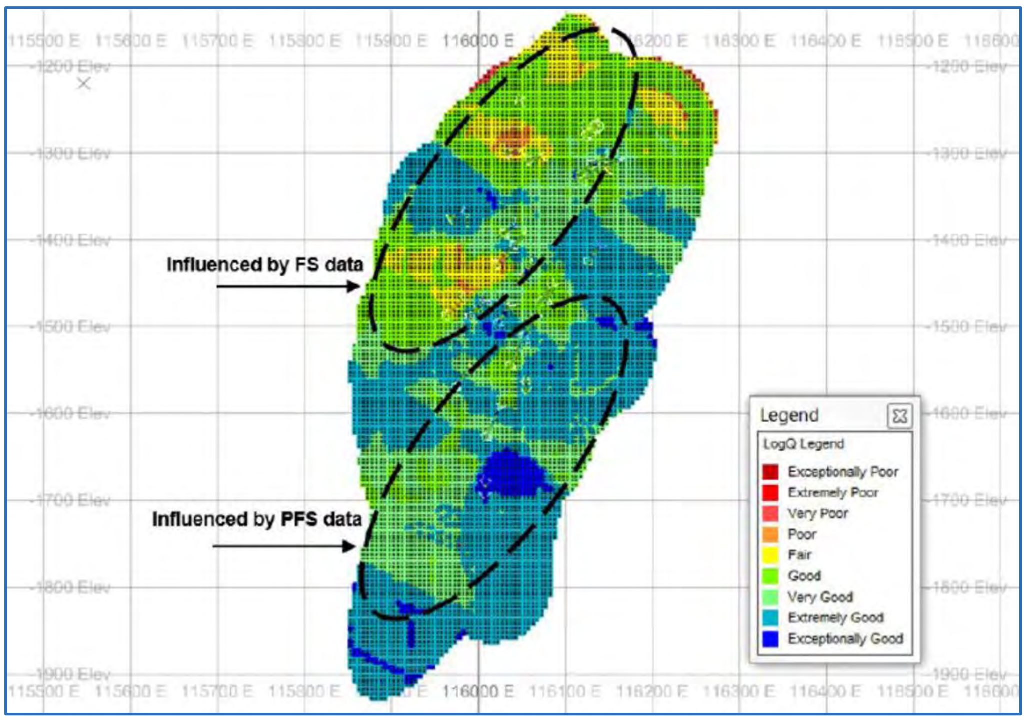
SRK, 2019

19) Plan View at 1,486 mL



SRK, 2019

b) North–South section looking 1,360 east



SRK, 2019

c) West–East section looking 194,345 North (Zone 3)

From the creation of the geotechnical block model it was observed that in general rock mass conditions within the vicinity of the BZ and SZ are good. There are areas however, where lower rock mass quality is evident. These are generally localised zones in the higher levels of the mine and in certain areas within the vicinity of the SZ. When using the block model, the following should be kept in mind:

- As the distance from the boreholes increases, the confidence in the block model decreases. The geotechnical block model, thus serves as a platform which can be built upon on a continuous basis as more data is gathered and as mining takes place.
- While the rock mass quality is generally lower within the vicinity of the SZ, the FS logging results are also more conservative compared with the SS and PFS logging results. The rock mass quality in the block model in areas where the FS holes are located (typically in the northern area of geotechnical zone 1 where the SZ is located) is, therefore, more conservative.
- As there is no logging data available for various regions in the footwall of the BZ, the block model could not be extended into this region. Caution must be exercised when developing through these areas, as it may be possible to intersect poor rock mass conditions on a local scale.

Overall the geotechnical block model serves as a platform which can be built upon on a continuous basis as more data is gathered and as mining takes place. As there is no data present in various regions in the footwall of the project area, it is recommended that boreholes are drilled in these locations to verify the quality of the rock mass.

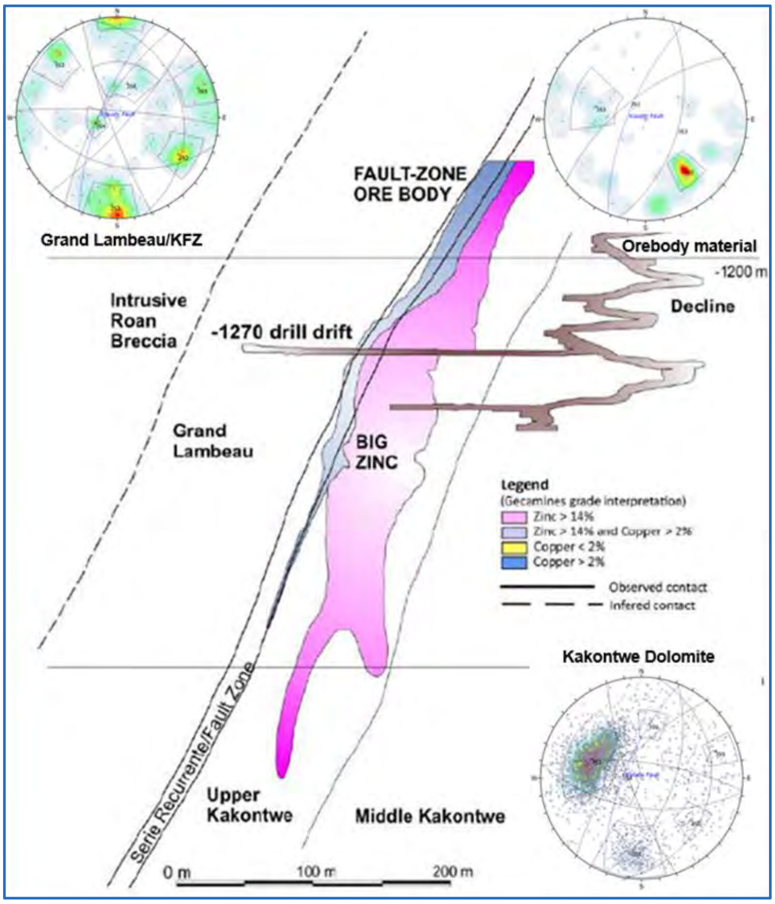
16.1.5.4 Structural Analysis

Based on the geology of the Kipushi region (Section 7), three geotechnical structural domains were initially outlined, which comprises the major lithological units of the project area (Table 16.22). Using this classification, the Rocscience software DIPS was utilised to plot joint orientation data for each domain. Note that joint orientation data was plotted for the mapping and borehole data separately and was then combined once trends were identified. As the stereographic projections showed that similar trends are present across all the major lithologies and domains (hanging wall, BZ and SZ orebody, and footwall), all the data was combined to produce a stereographic projection which represents the entire Kipushi area (Figure 16.7). A summary of the joint sets identified is presented in Table 16.23. The joint spacing determined is presented in Table 16.24.

Table 16.22 Kipushi – Structural Domains

Structural Domain	Major Lithologies	Mining Position
Kakontwe Dolomite	Upper Kakontwe dolomite (UK SDO) Middle Kakontwe dolomite (MK SDO)	Footwall and hanging wall (at lower depths of the orebody)
Grand Lambeau/Kipushi Fault Zone (GLB/KFZ)	Siltstone (SSL) Shale (SSH) Sandstone (SST)	Hanging wall
Orebody Material (Big Zinc)	Massive Brown Sphalerite (MBS) Massive Sulfides	Orebody

Figure 16.7 Identified Joint Sets per Structural Domain



SRK, 2019

Table 16.23 Joint Set Summary

Item	JS1	JS2	JS3
Mean Dip	75	73	55
Mean Dip Direction	004	301	113
Number of Joints	313	137	2,240

Table 16.24 True Joint Spacing (excluding closely spaced joint clusters)

Item	Unit	JS1	JS2	JS3
Minimum	m	0.5	0.5	0.5
Mean	m	4.4	5.7	3.4
Maximum	m	28.9	82.6	108.7
Standard Deviation		6	12	7

Based on the structural analysis conducted for the Kipushi Mine, three major joint sets (JS1, JS2, and JS3) have been identified across the project area. A summary of the results and recommendations are outlined as follows:

- Stereographic projections created for the GLB/KFZ (hanging wall), the Kakontwe dolomite (footwall), and the sphalerite (orebody) to determine the joints trends from mapping, ATV, and bore hole data. The data was of satisfactory quality and the stereographic projections showed that similar joint trends are present across all the major lithologies (GLB/KFZ, Kakontwe dolomite, and the sphalerite orebody). The BZ and SZ ore bodies comprise the sphalerite rock units with similar joint trends. All data was combined to produce a stereographic projection which represents the entire Kipushi Project area.
- JS1 is a dominant northerly dipping joint set that which represents the bedding of the sedimentary host rock.
- JS2 is a north-west dipping joint set present across the project area. This joint set is sub-parallel to the Kipushi Fault and the D2 deformation phase within the Lufilian Belt, which comprises west–south-west–east–north-east trending strike-slip tectonics.
- JS3 is a dominant south-west dipping joint set present across the project area. JS3 is particularly evident in the Kakontwe dolomite (footwall) and in the orebody material.
- JS2 and JS3 are interpreted as a conjugate set that can be related to the brittle tectonic stage 5. Stage 5 represents a late orogenetic extension to extensional collapse after D2 took place. As both these joint sets could be found in the GLB/KFZ (hanging wall), they postdate D2 which supports this interpretation.
- JS1, JS2, and JS3 are present across the project area. These joint sets were also identified in PFS structural analysis.

- JS4, JS5, and JS6 were identified as minor/random joint sets in the PFS, however, will not be included for further analyses in the FS. While it is believed that these minor joint sets do exist, these sets do not occur frequently and are likely to be localised joint sets (see SRK report no. 505390).
- Overall JS1 and JS3 form the most dominant joint sets in the Kipushi Project area. Based on the stereographic projections, JS3 appears more dominant that JS1, however, this is likely due to an orientation bias as the majority of the boreholes are orientated normal to JS3 and sub-parallel to JS1. Furthermore, the few boreholes that are located normal to JS1 were often not logged in detail (geological logs).
- Joints are predominantly rough planar, and in many cases are only stained, with the exception of JS3, which contains a large proportion of medium grained non-softening infill.
- The mean friction angles across the joint sets range from 39–54°.
- The majority of joints are >1 m in length and generally range from 1–10 m. It is likely that a number of joints are >10 m in length, however, this is difficult to determine due to the extent of exposure in the excavations.
- A block probabilistic joint analysis will be conducted using the software Jblock to determine the influence of the major joint sets on proposed stope orientations.
- While the structural analysis provides an impression of the major joint sets across the project area, further geotechnical scanline mapping should be conducted regularly as mining commences to allow for the identification of low angle joints in the hanging wall, localised joint sets, and for potential wedges and instabilities.
- Based on the structural analysis the mining stopes should be orientated between JS1 and JS3. Furthermore, stopes should not be located parallel to JS1 or JS3 as this is likely to result in slabbing of the sidewalls. These recommendations have been implemented in the BZ mine design.

16.1.6 Analysis and Design of Stopes

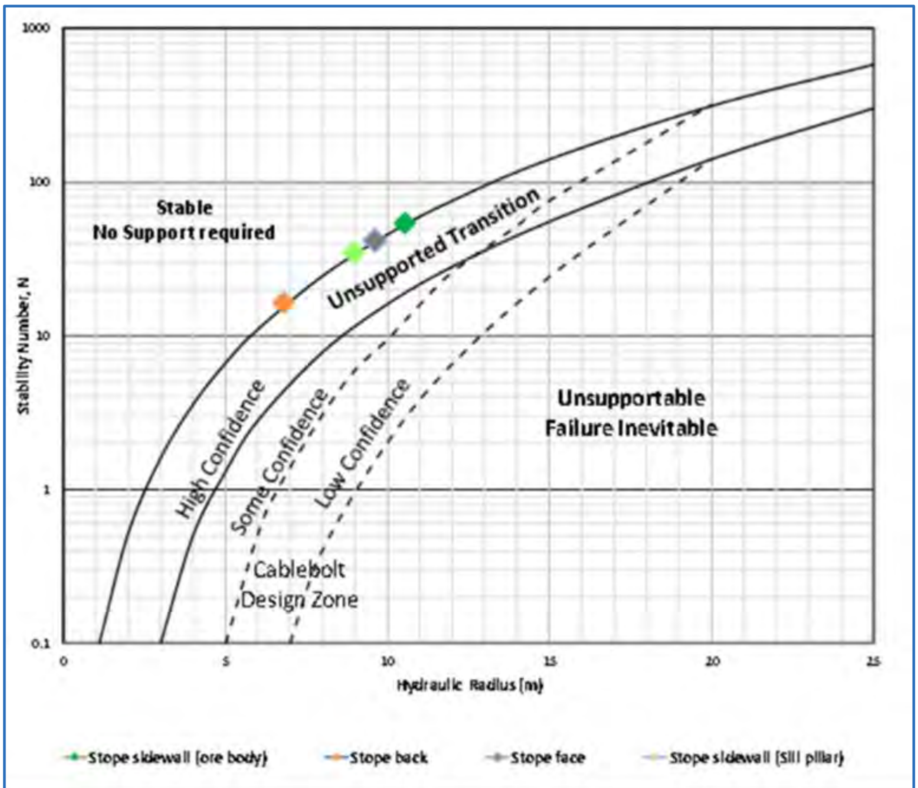
16.1.6.1 Empirical Stope Design

The stability of steeply dipping (greater than 60°) stopes were assessed using the stability method developed by Mathews et al. (1981) and modified by Potvin (1988), Potvin and Milne (1992) as presented by Hutchinson and Diederichs (1996).

To undertake the analysis, the stability number was determined from the rock mass classification carried out. The stability factors A, B, and C were obtained from the rock strength, stress analyses of the stopes and joint orientations in relation to the stope surfaces. Figure 16.8 and Figure 16.9 show the stability graph method input parameters and the resulting allowable HR for the stope walls located in the orebody and boundaries for the critical JS1 and JS3 for the BZ and SZ respectively.

It is important to note that these analyses are only applicable to secondary stopes if tight filling has been successfully achieved. Poor tight filling will result in less than optimal performance during mining of secondary stopes.

Figure 16.8 BZ – Stability Assessment of Stope Sidewalls



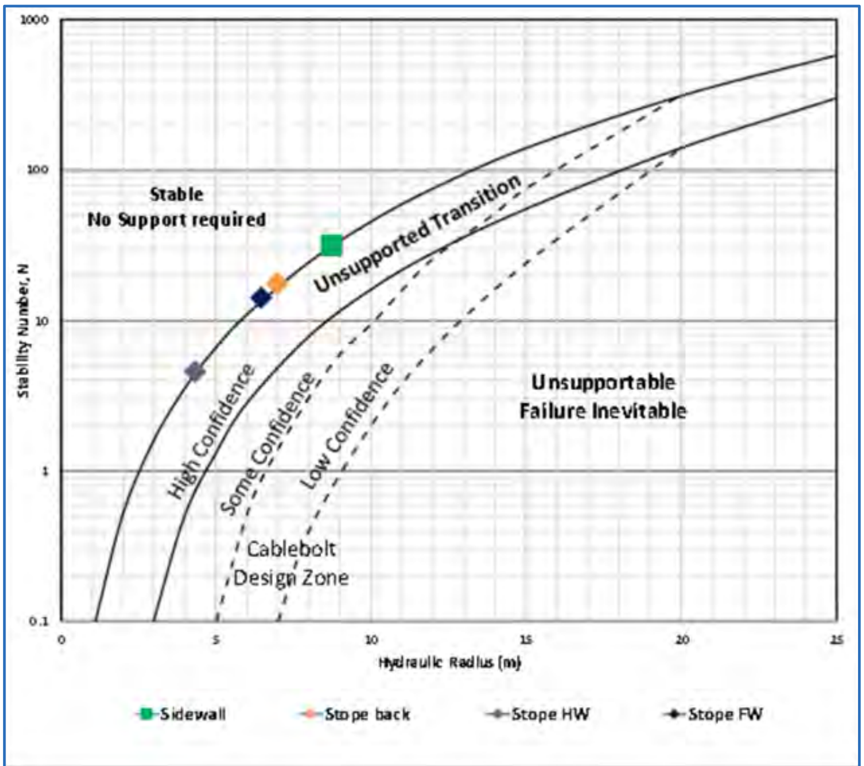
	Q'	A	B	C	N	HR
Stope sidewall	28	0.38	0.63	8.0	54	10.5
Stope back	28	0.42	0.70	2.0	16	6.8
Stope face	28	0.42	0.46	8.0	42	9.6
Stope sidewall (Sill pillar)	28	0.25	0.63	8.0	35	9.0

SRK, 2019

In the PFS study, the orientation of the stopes were optimised in relation to the joint set orientations, therefore, the BZ stope walls are favourably orientated. The allowable HR is greater than the required design of 6.0 m and 10.0 m for the stope back and the sidewalls respectively. However, the SZ orebody is narrow and there are limited options in terms of optimising the stope orientations. The design HR for the SZ transverse stopes is 5.0 m and 7.5 m for the stope back and sidewalls respectively. The transverse stopes are not adversely affected as the maximum unfilled stope length will be greater than the orebody thickness of 30 m. The allowable HR radius for the stope sidewall will be 8.7 m which satisfies the extraction of the stope to the maximum orebody thickness.

The longitudinal stopes are orientated in the strike direction which is parallel to the joint set 1 and the hanging wall and the footwall are unfavourably orientated. The allowable HR for the hanging wall will be 4.3 m which corresponds to 15 m unfilled stope length for 20 m high stope. Reducing the stope height to 15 m, increases the unfilled stope length to 20 m. Based on the Map3D model block the longitudinal stopes are approximately 16% by volume of the total SZ stopes. The SZ stopes will be extracted towards the end of the life-of-mine (LOM) and there will be enough time to plan the extraction of the narrow longitudinal stopes.

Figure 16.9 SZ – Stability Assessment of Stope Sidewalls



	Q'	A	B	C	N	HR
Stope face/sidewall	30	0.46	0.30	8.0	32	8.7
Stope back	30	0.42	0.70	2.0	18	7.0
Stope HW	11	0.42	0.20	5.0	5	4.3
Stope FW	11	0.81	0.20	8.0	14	6.5

SRK, 2019

Table 16.25 and Table 16.26 summarises the maximum unsupported stope dimensions and the allowable HR for the stope walls and orebody boundaries. Note that while the maximum HR per stope surface have been presented, the unsupported stope lengths are specific to practical considerations and the design HR, which is based on the stope dimensions as per the mine design parameters (Section 16.3.1).

Table 16.25 BZ – Maximum Stope Dimensions (unsupported)

Stope Surface	Maximum unsupported Length (m)	Width (m)	Height (m)	Design HR (m)	Allowable HR (m)
Stope back (roof)	60	15	–	6.0	6.8
Stope sidewall	60	–	30	10.0	10.5
Stope face	–	15	30	5.0	9.6
Stope sidewall (sill pillar)	60	–	15	5.0	9.6

Table 16.26 SZ – Maximum Stope Dimensions (unsupported)

Stope Surface	Maximum unsupported Length (m)	Width (m)	Height (m)	Design HR (m)	Allowable HR (m)
Stope back (roof)	30	15	–	5.0	7.0
Stope sidewall	30	–	30	7.5	8.7
Stope HW	30	–	20	6.0	4.3
Stope FW	30	–	20	6.0	6.5
Stope face	–	15	20	4.3	8.7

16.1.6.2 Block Analysis

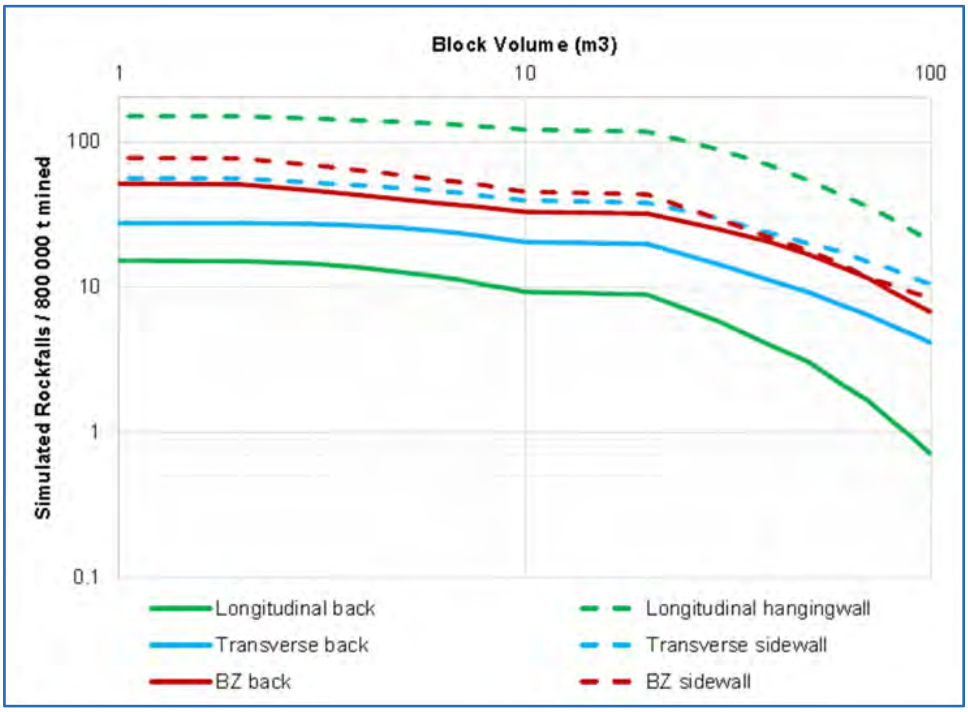
A risk assessment of the failure potential of the back area and side walls of the proposed stopes for the BZ and the SZ were carried out using Jblock computer software. Jblock is designed to create and analyse geometric blocks or wedges known as blocks.
The programme has the capacity to simulate a large number of blocks as a function of joint set characteristics in relation to the excavation orientation and is used to derive a statistical failure distribution.

The results of the block analysis are summarised in Table 16.27 and is presented in Figure 16.10 for the BZ and SZ. The results were normalised to rockfalls per 800,000 t mined per year which is approximately the production target for Kipushi. Table 16.27 shows that the overall expected dilution by volume due to the influence of the joints is low and is unlikely to influence the overall production. The expected percentage dilution was determined from the simulated volume of failed rockfalls to the total volume of simulated blocks. Note that the expected dilution for the BZ and SZ transverse stope sidewalls were added. The SZ longitudinal hanging wall stope dilution stand out and the wall will be in waste while the other stope surfaces will be in ore.

Table 16.27 Jblock Results Summary

Stope Surface	Surface Area (m^2)	Hydraulic Radius	Expected Dilution (%)
Longitudinal back	150	3.0	0.2%
Longitudinal hanging wall	300	4.3	3.8%
Transverse back	450	5.0	2.8%
Transverse sidewalls	900	7.5	
BZ back	900	6.0	2.6%
BZ sidewall	1,800	10.0	

Figure 16.10 Jblock Results for the BZ and SZ Stope Surfaces



SRK, 2019

Figure 16.10 shows the distribution of the cumulative volume of simulated rockfalls per 800,000 t mined per year. Note that the simulated rockfalls per 800,000 t mined per year for the BZ and SZ transverse stopes was added. It was assumed that a simulated rockfall approximately 20 m^3 is likely to cause significant damage to equipment and could affect production and was used as a criterion in the analysis. The SZ longitudinal stope back shows that nine rockfalls are likely to occur per 800,000 t mined per year. This is because the stope back has the least HR and approximately 70% of the surface is supported.

The number of simulated 20 m^3 rockfalls increases for the other stope surfaces as follows:

- BZ back – 32 rockfalls per 800,000 t.
- BS sidewalls – 43 rockfalls per 800,000 t.
- SZ transverse back – 20 rockfalls per 800,000 t.
- SZ transverse sidewalls – 38 rockfalls per 800,000 t.
- SZ longitudinal back – 9 rockfalls per 800,000 t.
- SZ longitudinal hanging wall – 116 rockfalls per 800,000 t.
- SZ longitudinal hanging wall – 116 rockfalls per 800,000 t.

The SZ longitudinal hanging wall has the most simulated rockfalls per 800,000 t, this is because the stope surface is unfavourably orientated (near parallel on strike) in relation to JS1. The longitudinal hanging wall will be the most problematic wall even after reducing the stope length to 15 m as indicated by the empirical stope design. The results are relatively comparable to the empirical stope assessment in determining the stable stope span (Section 16.1.6.1). Note that JS2 is not prevalent but was treated as if it exists everywhere in the rockfall analysis.

The SZ transverse and the BZ stope sidewalls have a similar number of simulated rockfalls (38 and 43 rockfalls per 800,000 t). However, the stope backs show that more rockfalls are likely to occur in the BZ than the SZ transverse stope. The results are relatively comparative with the empirical stope design (Table 16.25 and Table 16.26) as indicated by the allowable HR (6.8 m and 7.0 m for BZ and SZ to respectively).

16.1.6.3 Backfill and Bulkhead Assessment

The extraction of primary and secondary stopes requires the use of backfill. The placement of backfill in the stopes provides the following functions:

- Support of the stope surfaces and maintains the hydraulic radius within design limits.
- Prevents rockfalls and caving.
- Provides a working platform for the upper stopes.
- Provides protection cover for the extraction of the primary sill stopes.

Backfill will require to be free standing and to provide protection cover during undermining.

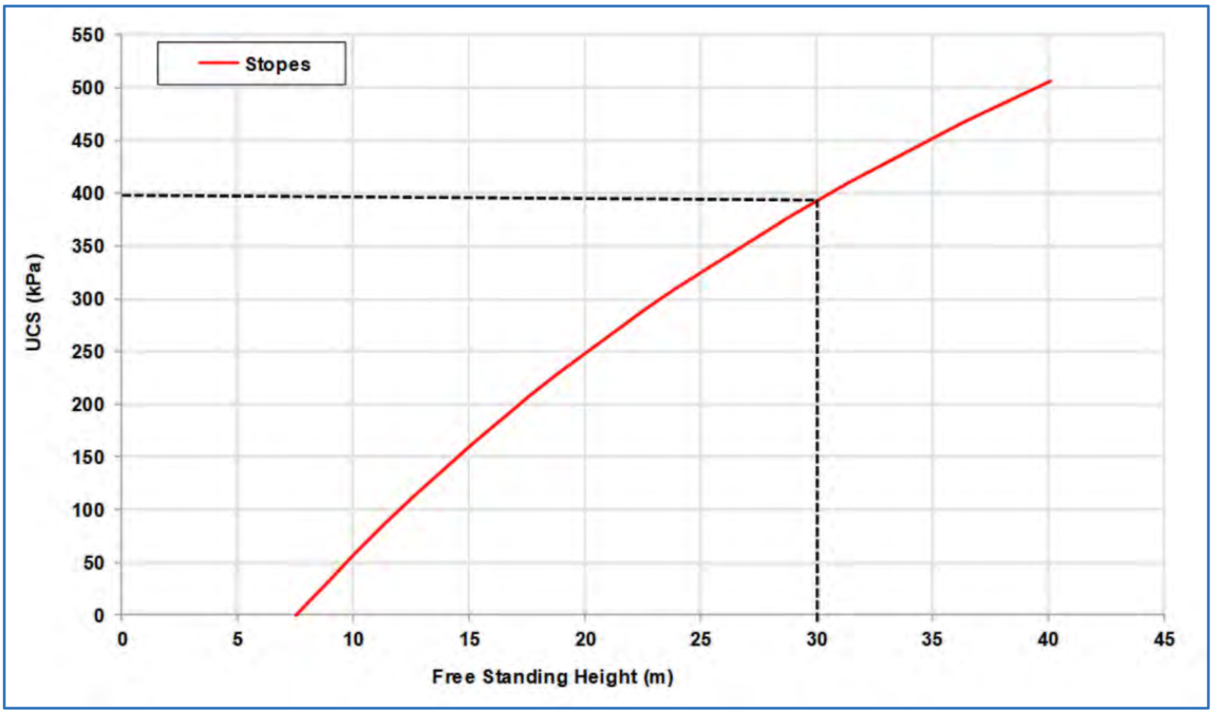
Backfill Strength Requirements

Kipushi intends to use a Crushed Rock Fill (Crushed RF) with additives. The purpose of the additives is to develop cohesion within the backfill particles so that the exposed fill surfaces are self-supporting when the adjacent stopes are extracted. The backfill strength requirements were determined using an analytical method for static analysis of backfill as described by Mitchell et al. (1981). The analytical method assumes that only one sidewall face will be exposed and ignores lateral pressure due to the surrounding rock surfaces. During the extraction of the secondary stopes, the backfill in the primary stopes will be exposed on one side for the full height of 30 m. It is necessary to maintain a stable backfill free face during extraction.

Figure 16.11 shows that the required free-standing backfill strength for the upper primary stopes will be 400 kPa. The lower and upper secondary stope sidewalls will not be exposed, however, the face needs to be designed as free standing to prevent backfill collapsing onto the broken ore where stope lengths are greater than 60 m. Figure 16.12 shows that the free-standing strength requirement of 220 kPa will be required for the secondary stope lengths greater than 60 m. The lower and upper secondary stopes with a total length less than 60 m will not require free standing backfill strength requirements as the backfill face will not be exposed.

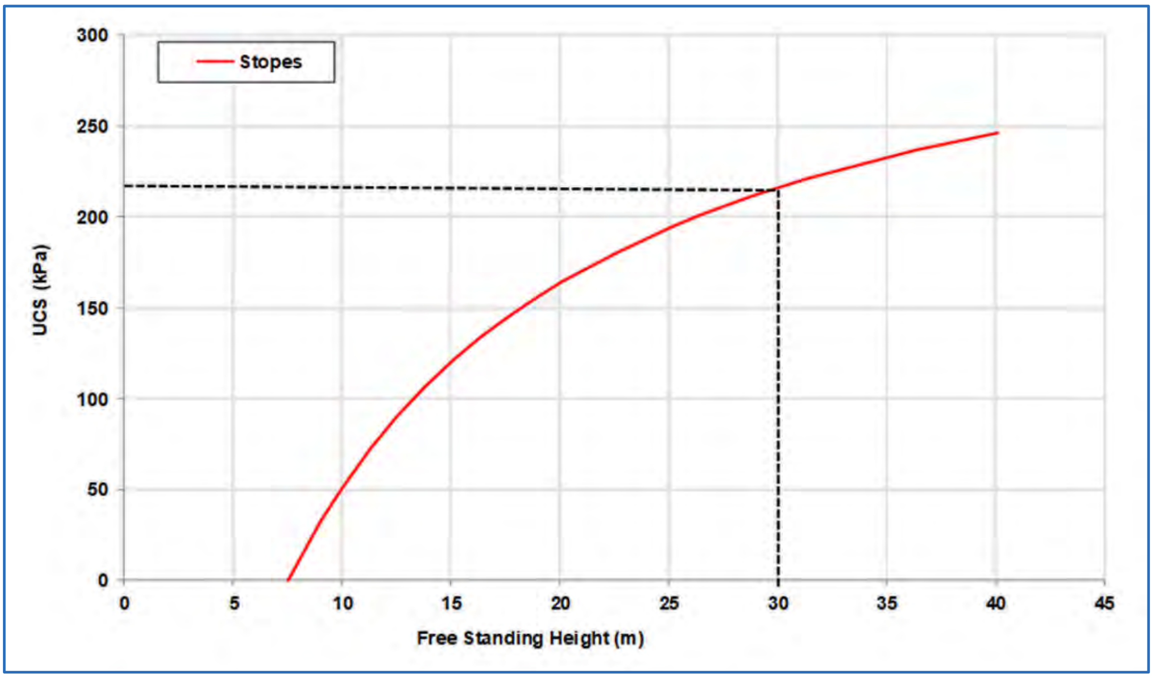
The SZ transverse stopes both primary and secondary will require 310 kPa backfill strength to be free standing (Figure 16.13). The secondary which are less than 30 m long will not require free standing backfill strength requirements. The longitudinal stopes will require 220 kPa backfill strength to avoid fill collapse onto the blasted ore.

Figure 16.11 BZ – Backfill Free-Standing Strength Requirements for Upper Primary Stopes



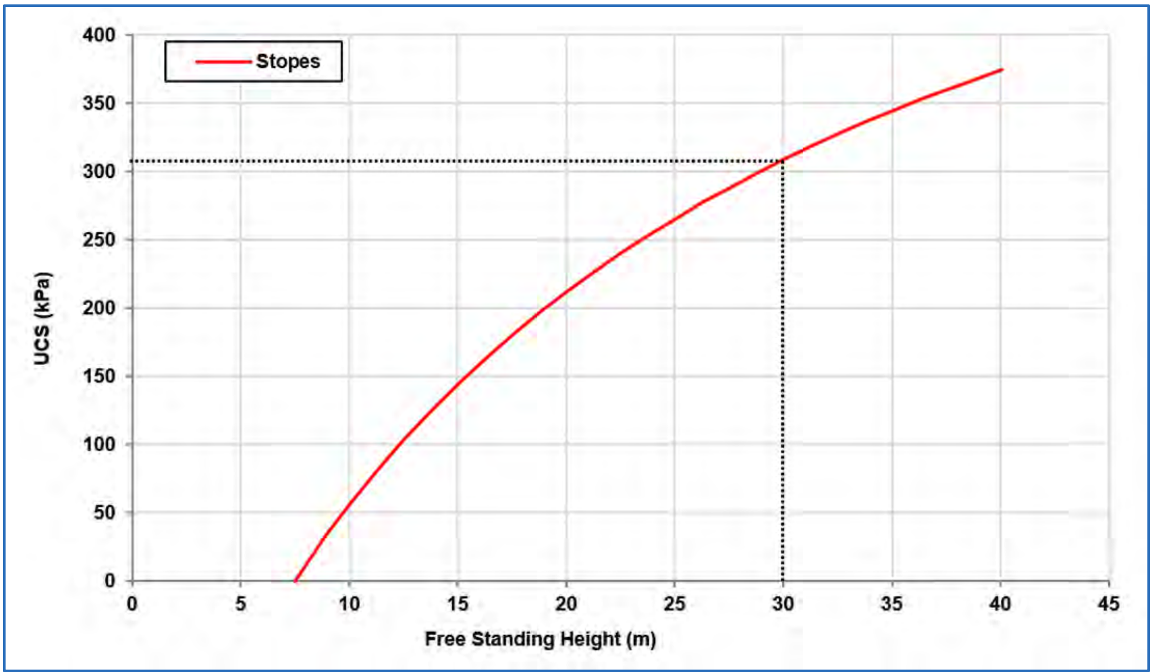
SRK, 2019

Figure 16.12 BZ – Backfill Free-Standing Strength Requirements for the Lower and Upper Secondary Stope Faces



SRK, 2019

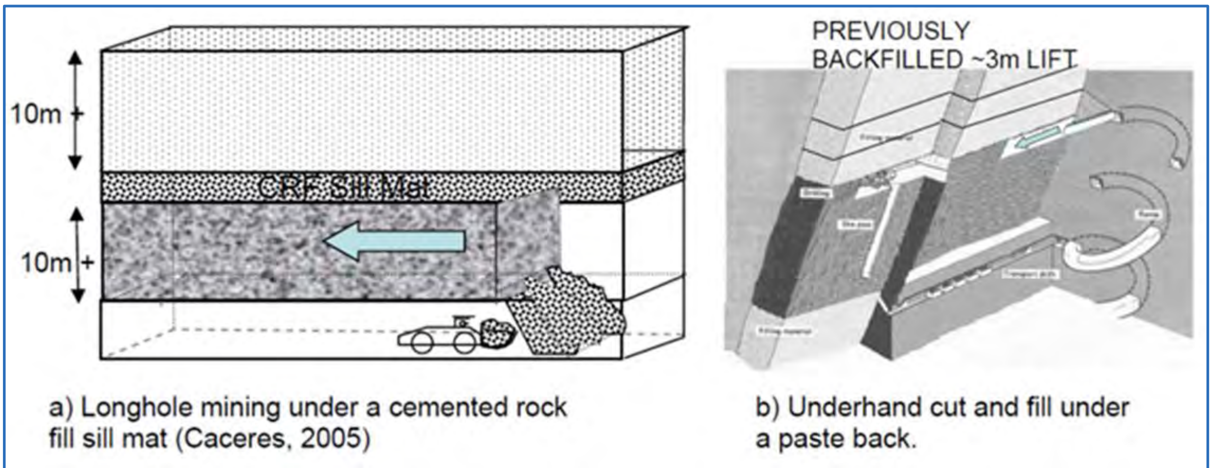
Figure 16.13 SZ – Backfill Free-Standing Strength Requirements for the Primary and Secondary Transverse Stope Faces



SRK, 2019

The extraction of the BZ sill stopes will require undercutting the backfill in the lower stopes (Figure 16.14). Only primary stopes will be extracted. The BZ sill stope cross-sectional dimensions are 15 m W x 15 m H. The sill pillar will comprise primary and secondary stopes. Note that only primary stopes will be extracted. Secondary sill pillars will remain as permanent pillars.

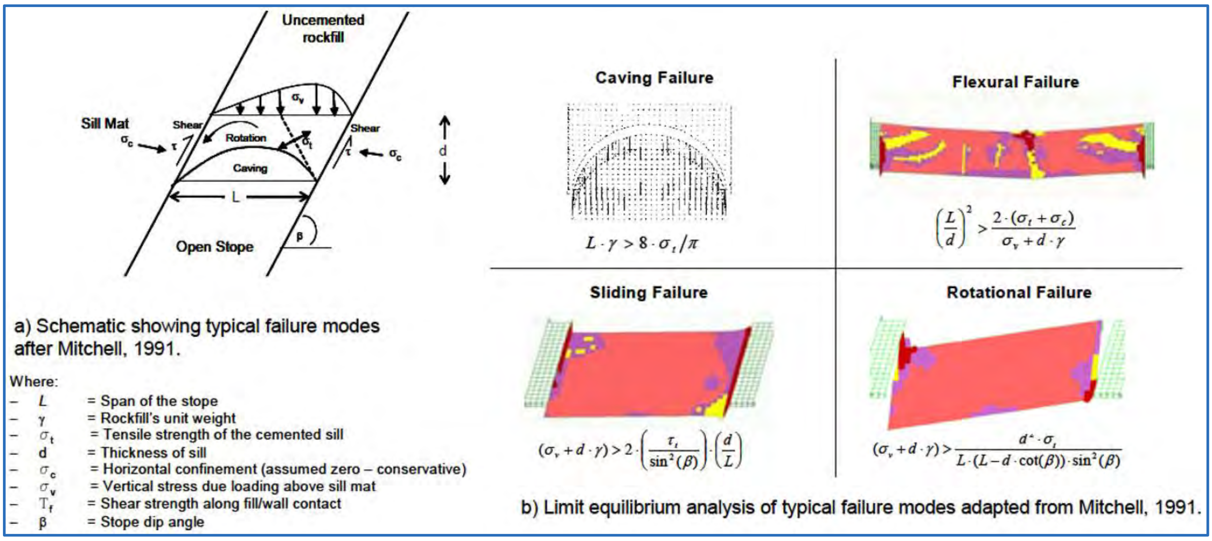
Figure 16.14 Example of Undercutting Backfill



Pakalnis, 2005

Backfill strengths are required to be designed to take into consideration undercutting. The method proposed by Mitchell, (1991), which is based on limit equilibrium analysis for four failure modes, namely caving, flexural, sliding, and rotational failure (Figure 16.15), was used to assess backfill sill stability. In this assessment, it was assumed that continuous placement of the fill over the full height of the stope (30 m) would be ensured, preventing the formation of cold joints. Backfill placement and monitoring are, therefore, critical. The coarse aggregate in the cemented rock fill (CRF) should improve the shear strength of the cold joints, but this will need to be tested during the execution stage.

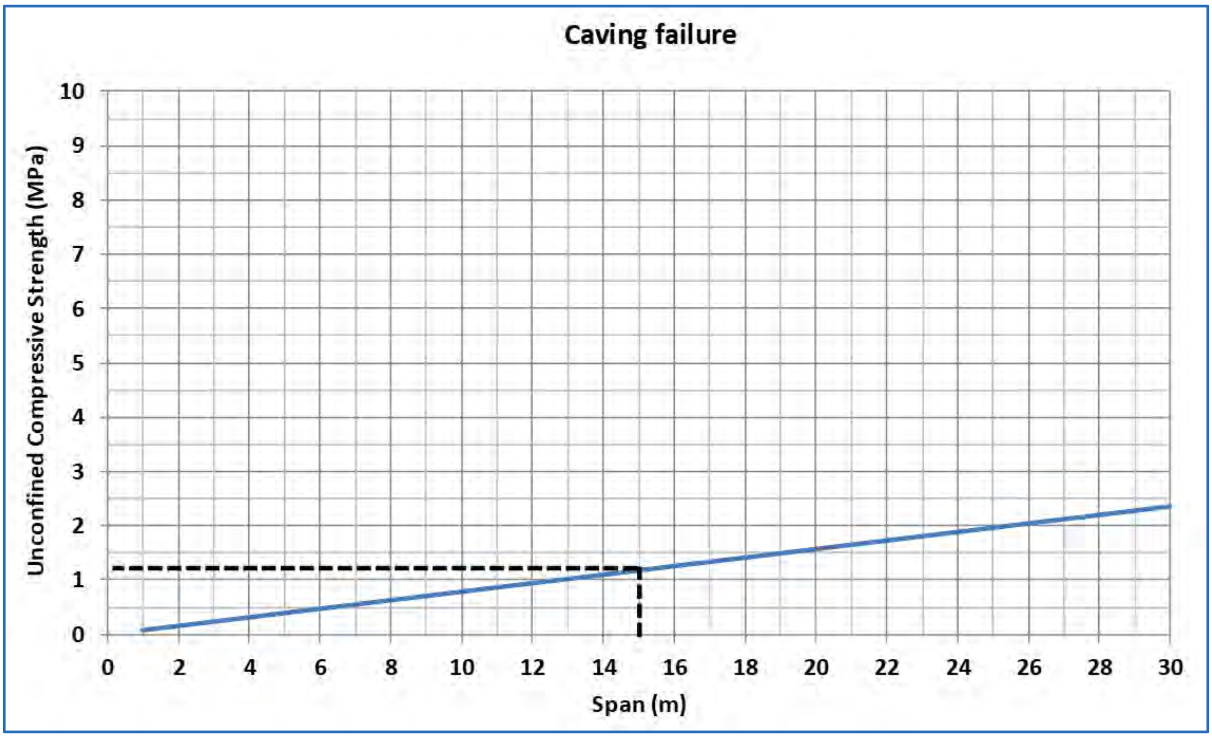
Figure 16.15 Limit Equilibrium Criteria



Mitchell, 1991; after Pakalnis et. Al. 2005

Caving failure (Figure 16.16) was found to be the critical failure mode for a 30 m thick backfill sill requiring a minimum sill strength of 1.2 Mpa. Note that there will be no backfill undercutting on the SZ, therefore, backfill sill strengths are not required.

Figure 16.16 Stability Chart for Caving Failure



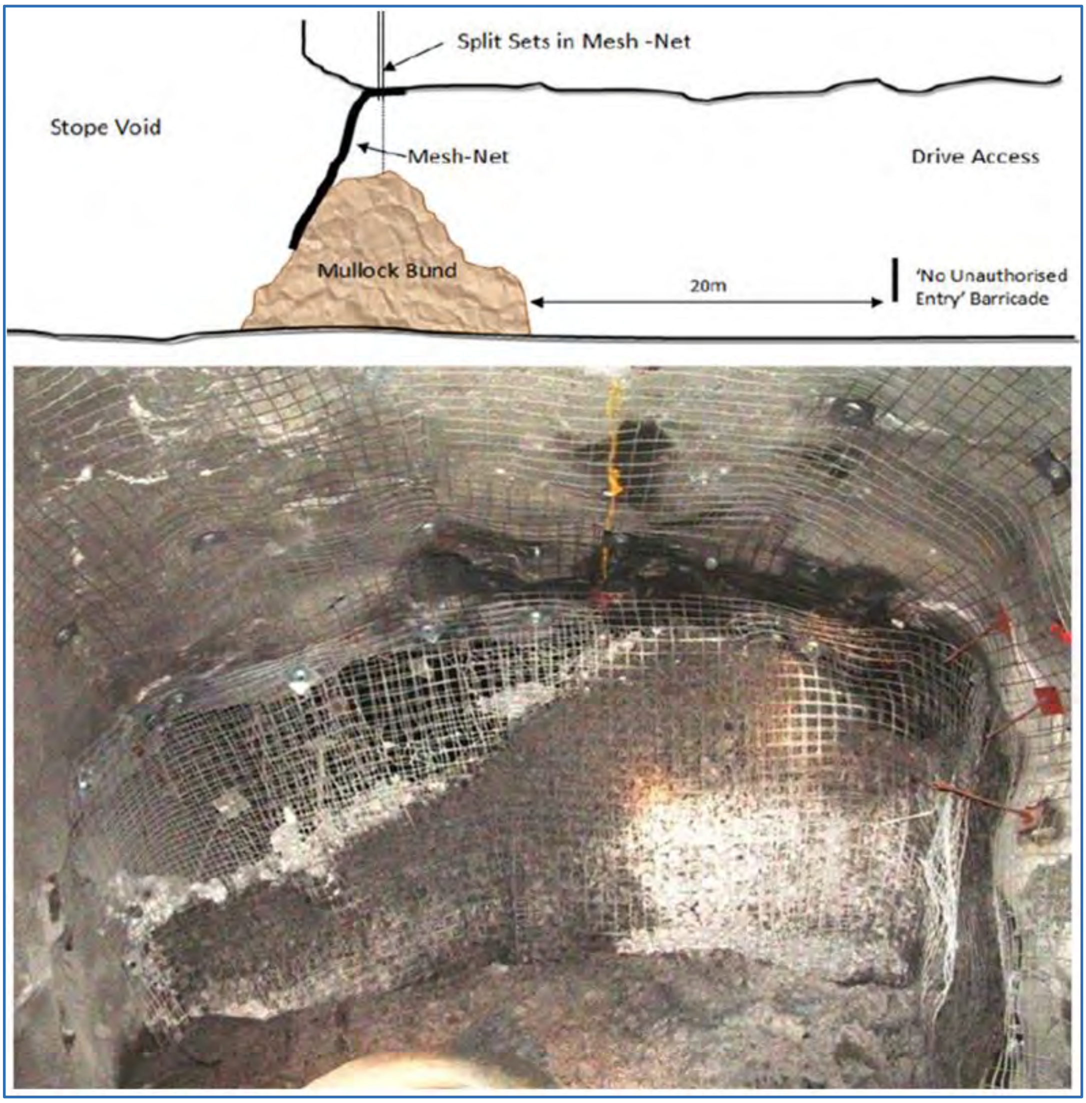
SRK, 2019

Bulkheads

Waste rock barricades were proposed for Kipushi to contain unconsolidated CRF. The method was reviewed by SRK and found to be acceptable.

The construction of the waste rock barricade (WRB) is illustrated in Figure 16.17. Note that mesh is installed in the roof and sidewalls using split sets prior to dumping the waste rock against the mesh. The installation of the mesh is to improve shear resistance of the WRB. Note that the initial pour for all stopes will require binder to strengthen the WRB and this includes the secondary stopes that do not require backfill free standing strength. CRF containing binder at approximately 92% solids (by weight) is planned for use at the Kipushi Mine, which significantly reduces the loading on the WRB.

Figure 16.17 Construction of Waste Rock Barricade



Mining Plus, 2019

16.1.7 Support Requirements and Assessment of Anticipated Ground Conditions

16.1.7.1 Support Requirements

Support recommendations are based on underground observations, discussions held with the mine personnel, numerical modelling, and Q ratings determined for the orebody and footwall. The recommendations were based on the discussions held with Mr Marcin Szpak from Kipushi Corporation SA (KICO) (South Africa) and SRK personnel. Rock mass classification will be used where stress influence is insignificant, that is where Q>20 good ground condition S0 support will be required. Where Q<20 indicating relatively fair to poor ground conditions S1 support will be required. The Q rating support recommendation will be applied to the decline and level access development.

The potential damage on the decline, access drives and stope drives were assessed using the Depth of Fracturing (DF) Martin, et al. (1999) and Cai and Kaiser, (2014). The DF is an empirical method which relates excavation damage to the maximum tangential stress determined from numerical modelling and takes into consideration the excavation size.

The DF was used to determine the type of support required to maintain stability. It is recommended that DF (extent of stress driven failure) be verified with field observations during the construction of the excavations or use of borehole cameras in the case of raisebore.

Table 16.28 shows the support recommendations based on Q ratings and elastic modelling whereby DF is calculated.

Table 16.28 Support Requirements Based on DF

DF (m)	Damage Expected	Support
0 < DF	No fracturing	S0
0 < DF < 0.5	Minimum wall fracturing	S1
0.5< DF< 1.5	Stress damage, bulking and dynamic loading	S2
DF>1.5	Severe stress damage and dynamic loading	S3

The support requirements for the various excavations are presented in Table 16.29 and Table 16.30. Support specifications are presented in Table 16.31. Note that where the ground conditions are poor (Q<4) (8% based on current data), development support should be installed as specified in the development design parameters.

Table 16.29 Support Requirements per Excavation

Excavation	Level		Excavation Sizes	Support Required
Decline (long term)	1,200	1,440	5.3 m W x 5.8 m H	S0
	1,440	1,600	5.3 m W x 5.8 m H	S1
Level access (medium term)	1,200	1,395	5.4 m W x 5.8 m H	S0
	1,395	1,600	5.4 m W x 5.8 m H	S1
Stope and parallel access (medium term)	1,200	1,395	5.3 m W x 5.8 m H	S1
	1,395	1,600	5.3 m W x 5.8 m H	S2
Ore drives (lower)	1,200	1,600	5.5 m W x 5.0 m H	S2
Ore drives (upper)	1,200	1,600	6.0 m W x 6.0 m H	S2
Sill Pillar Stope drives – Primary	1,245	1,600	6.0 m W x 6.0 m H	S3
Sill Pillar Stope drives – Secondary (no re-entry into these stope drives)	1,245	1,600	8.0 m W x 6.0 m H	S2
Support of all intersections				L1
Support of large excavations (hoist, crusher chambers)				L2
Faults and shear zones				L1
Permanent working areas (workshops and magazines, main facilities, escape chambers) up to 8.0 m W x 6.0 m H				S1a
Stope brow support or where poor ground conditions are anticipated or poor blasting.				L3
Ventilation raises and intake ventilation raises (raise-boring)				L4
Old tunnels – extra support required where Q<4 or influenced by stress.				S2
Old large excavations (hoist, crusher chambers, workshops)				L2

Table 16.30 Support Standards for Excavations

Support	Area of Application	Support Standard
	Shafts (blind sink)	Primary support: Minimum 1.8 m long splits sets at 1.0 m x 1.5 m pattern with mesh. Secondary support: 300 mm concrete lining.
	Vent shafts (Raisebore) (high stress)	Minimum 50 mm shotcrete lining or concrete lining.
S0	Decline. No Fracturing. Access drive support for normal conditions (No fracturing). Geological structures.	2.4 m long tensioned full column resin rebar in a 2.0 m x 2.0 m pattern.
S1	Decline. Fracture depth <0.5 m. Access Drive. Fracture depth <0.5 m.	2.4 m long tensioned full column resin rebar in a 2.0 m x 2.0 m pattern with mesh on the roof down to 1.5 m above the floor.
S1a	Access Drive (Fracture depth between 0.5 m and 1.5 m, stress damage, bulking and high stress). Permanent working areas (workshops / magazines, main facilities, escape chambers) up to 8.0 m W x 6.0 m H.	2.4 m long tensioned resin grouted bars in a 1.5 m x 1.5 m pattern with mesh on the roof down to 1.5 m above the floor.
S2	Access Drive (Fracture depth between 0.5 m and 1.5 m, stress damage, bulking and dynamic loading). Stope drive support (Fracture depth between 0.5 m and 1.5 m, stress damage, bulking and dynamic loading). Stope drive (sill) support for secondary stopes (there will be no re-entry into these stope drives). Old tunnels – extra support required where Q<4 or influenced by stress.	3 m long tensioned resin grouted yielding bars in a 1.5 m x 1.5 m pattern with mesh on the roof down to 1.0 m above the floor.
S3	Stope drive (sill) support for primary stopes (Fracture depth >1.5 m, Severe stress damage and dynamic loading).	3.0 m long tensioned resin yielding bar in a 1.0 m x 1.0 m pattern with mesh on the roof down to 1.0 m above the floor.
L1	Support of intersections.	Primary support + Secondary support: minimum five 4.5 m long pre-tensioned, grouted cable anchors installed on the roof at the time of development.
L1	Faults and Shear Zones.	Primary support + Secondary support: minimum five 4.5 m long pre-tensioned, grouted cable anchors installed on the roof at the time of development + straps.
L2	Support of large excavations (hoist chambers).	Primary support + pre-tensioned, grouted cable bolts (minimum length= half excavation span), maximum spacing = 0.5 x bolt length with mesh.
L3	Stope brow support (where necessary).	Primary support + three rows (1.0 m apart) of three 6.0 m long grouted, cable anchors installed within 1.0 m of planned brow position.

Table 16.31 Support Specifications

Support Type	Support Specification
Splits sets (SS 33)	Outer diameter 33.5 mm to 34.2 mm, yield strength 420 Mpa black SUPRAFORM steel, minimum steel thickness 2.3 mm, hole size 30 mm to 32 mm.
Resin bolt	Rebar – Yield strength 500 Mpa black steel, 25–28 mm hole diameter, 20 mm bolt diameter. Capsule resin – Two component urethane silicate resin capsules. Fast <30 sec and slow 5–10 min setting time. Injection resin – Two component urethane silicate injection resin with water sealing properties.
Yielding bar (for S2 and S3 support)	Yield strength 500 Mpa black steel, 25–28 mm hole diameter, 20 mm bolt diameter, minimum energy absorption 30 kJ within 300 mm tunnel deformation.
Cable bolt and grout	Anchor – Minimum 18 mm diameter black steel, 380 kN ultimate load. Grout – Minimum 40 Mpa Ordinary Portland Cement, water to cement ratio 0.35:0.40.
Mesh	Black welded mesh, minimum 5 mm gauge, maximum 100 m aperture.
TSL membranes	Fastcrete M.

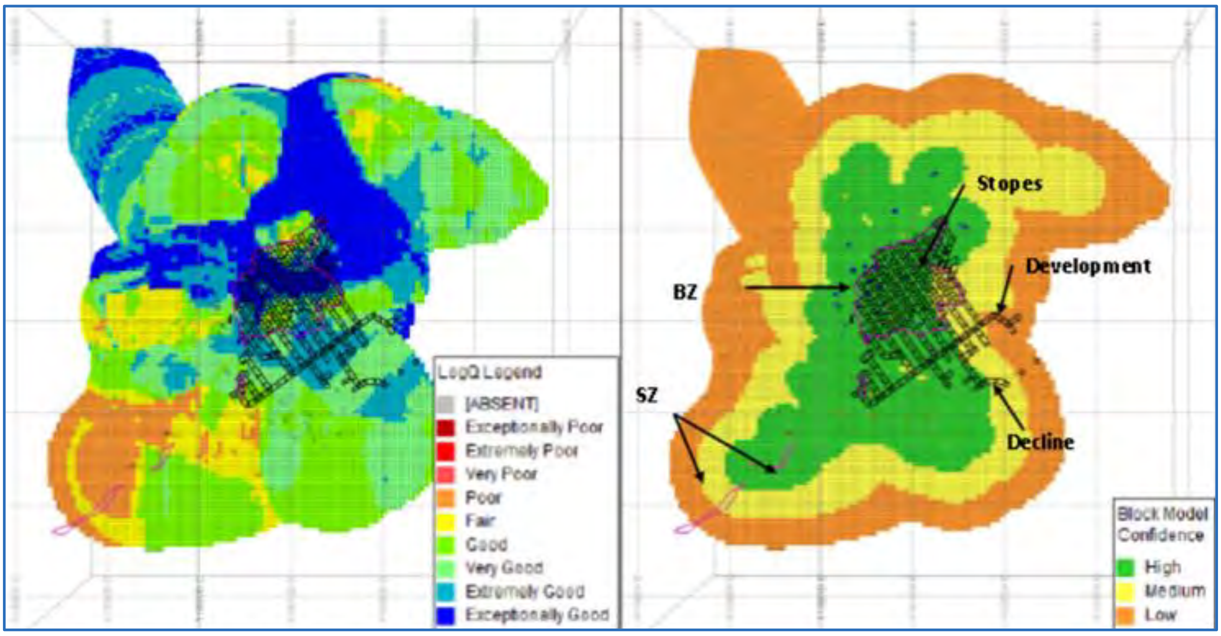
16.1.7.2 Assessment of Rock Mass Quality in Development

To assess the rock mass conditions that may be encountered within the planned developments, a geotechnical block model was created. Note that the geotechnical block model only provides an impression of what the conditions may be like and can provide an indication of where poor conditions may exist. Rock mass conditions will be verified once the rock is exposed during the excavation stage of the project.

Figure 16.18 illustrates an example of a plan view of the development depicted in the geotechnical block model. Block model confidence is indicated by three colours as follows:

- Green – High confidence
- Yellow – Medium confidence
- Orange – Low confidence

Figure 16.18 Rock Mass Quality and Geotechnical Block Model Confidence (1,395 mL)

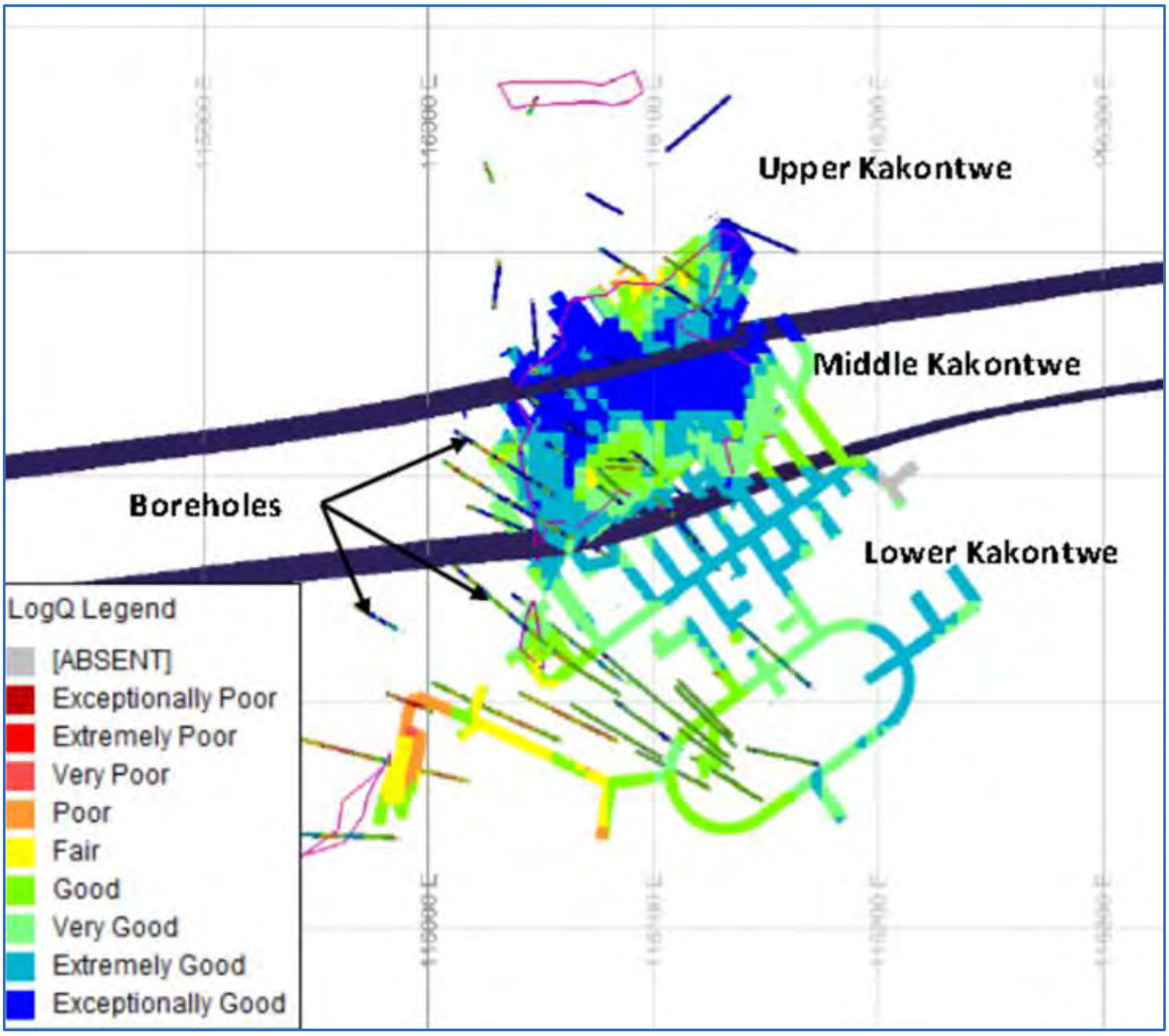


SRK, 2019; The black strings refer to the development while the purple colour outline refers to the orebodies. The rock mass quality results are indicated on the left while the block model confidence is indicated on the right. The colours used in the block model range from cold to hot whereby the cooler colours represent good to exceptionally good quality rock while the warmer colours refer to fair to extremely poor rock. Light grey colours indicate points that are outside the block model.

In the images that follow, the block model results and the confidence in the block model have been overlaid onto the developments. Boreholes containing the rock mass quality is also included.

The boundaries between the upper, middle, and lower Kakontwe dolomite in relation to the development and the BZ and SZ orebodies is indicated in Figure 16.19. Note that in general the best quality rock may be found within the middle Kakontwe dolomite where the BZ is located. The rock mass in the upper Kakontwe and lower Kakontwe dolomite is of a lower quality compared with the middle Kakontwe dolomite. This is since these formations are less massive and contain more fracturing. There is also a presence of chert in the upper Kakontwe and haematite staining in the lower Kakontwe which give rise to lower rock mass ratings. Furthermore, conservative logging in the lower Kakontwe dolomite (in the vicinity of the SZ) have led to more conservative results i.e. lower rock mass ratings.

Figure 16.19 Boundaries between the Upper, Middle, and Lower Kakontwe Dolomite in Relation to the Development



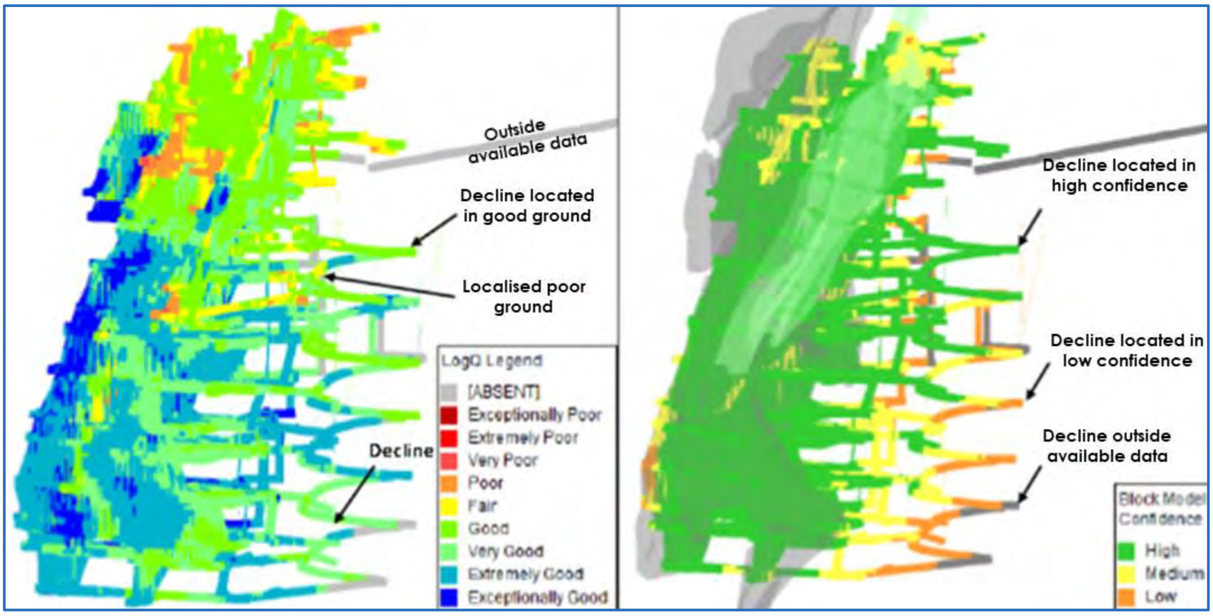
SRK, 2019

Decline

The rock mass quality and block model confidence along the decline is presented in Figure 16.20. In general, the block model indicates that the decline will be developed in good to extremely good rock mass conditions. In these areas, S0 support may be used (Q>20). There is a localised area that is located in good to fair rock. In these areas S1 support will be required (Q<20).

Please note that the block model confidence decreases in the decline as you move further away from the stopes. There are also portions of the decline that are located outside of the block model as there is no data available in these areas (indicated in grey). The rock mass quality should be verified as the excavation progresses by ongoing mapping and inspections.

Figure 16.20 Rock Mass Quality and Geotechnical Block Model Confidence along the Decline (Looking North-East)



SRK, 2019

Level Access and Parallel Access Drives

Examples of areas where lower rock mass conditions may be encountered on the various levels are summarised in Table 16.32 are presented from Figure 16.21 to Figure 16.27. Areas where low confidence in the block model exists, have also been highlighted.

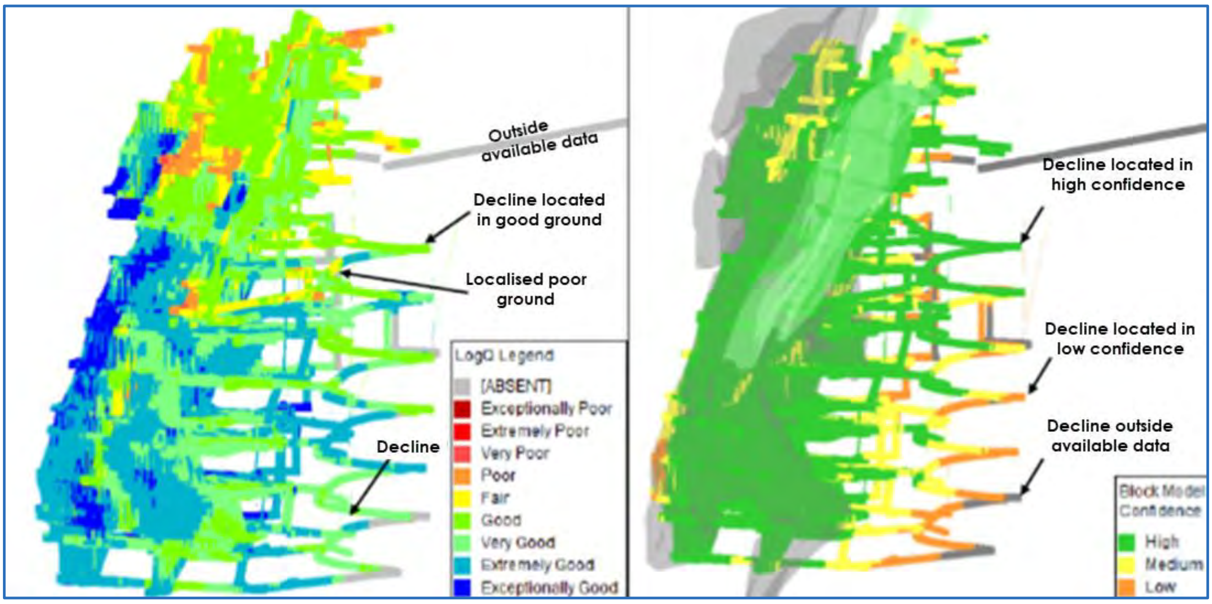
In general, the block model confidence is lowest on the excavations located on the eastern portion of the BZ orebody (see Figure 16.21 to Figure 16.27). Confidence in the block model are generally high within the SZ orebody. Note that while localised areas of poorer rock mass conditions are indicated on various levels of the SZ, this may be exaggerated by the conservative logging undertaken in this area. Rock mass conditions should be verified by continuous mapping and inspections during construction.

Table 16.32 Rock Mass Quality and Block Model Confidence along Selected Levels

Level	Rock Mass Quality		Block Model Confidence	
	BZ	SZ	BZ	SZ
1,220	Good	Poor and fair (localised)	High to low	High
1,290	Poor and fair (localised)	Poor and fair (localised)	High to low	High
1,395	Good to ext. good	Poor and fair (localised)	High to low	High
1,440	Good to ext. good	–	High to low	–
1,470	Good to exc. Good	–	High to low	–
1,515	Good to exc. Good	–	High to low	–
1,560	Good to exc. Good	–	High to low	–

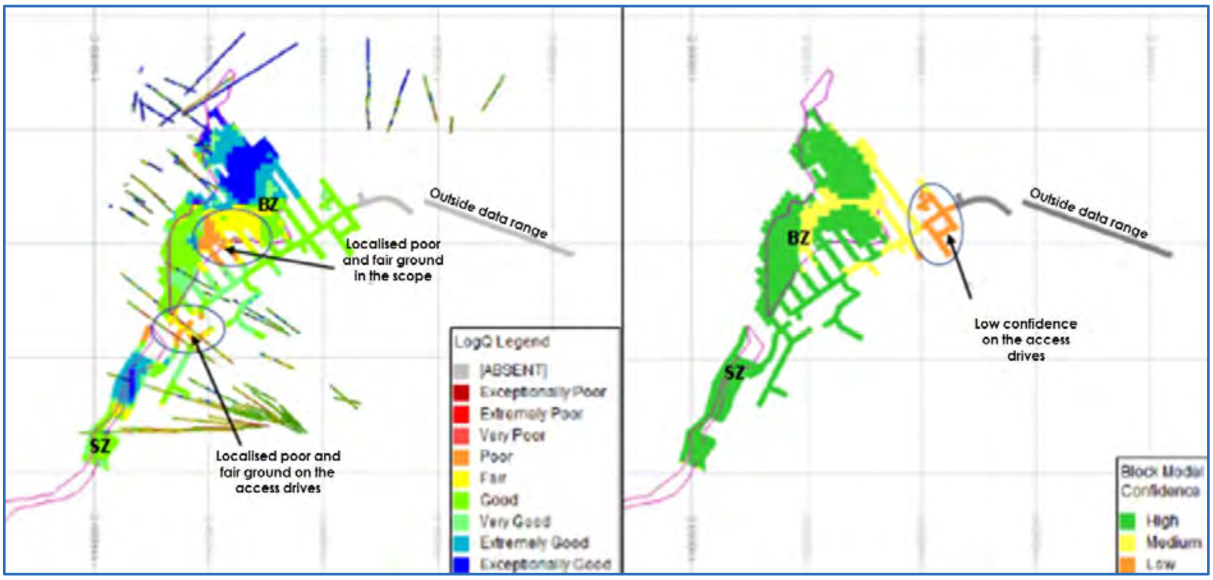
ext. = extremely
exc. = exceptionally

Figure 16.21 1,220 mL (Plan View): Rock Mass Quality and Block Model Confidence



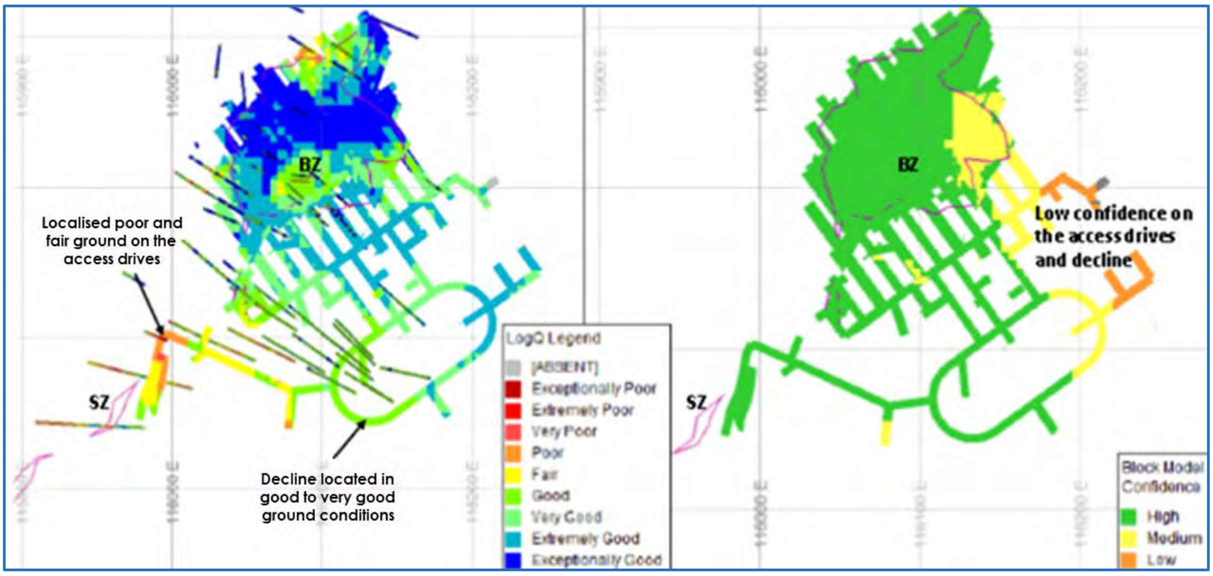
SRK, 2019

Figure 16.22 1,290 mL (Plan View): Rock Mass Quality and Block Model Confidence



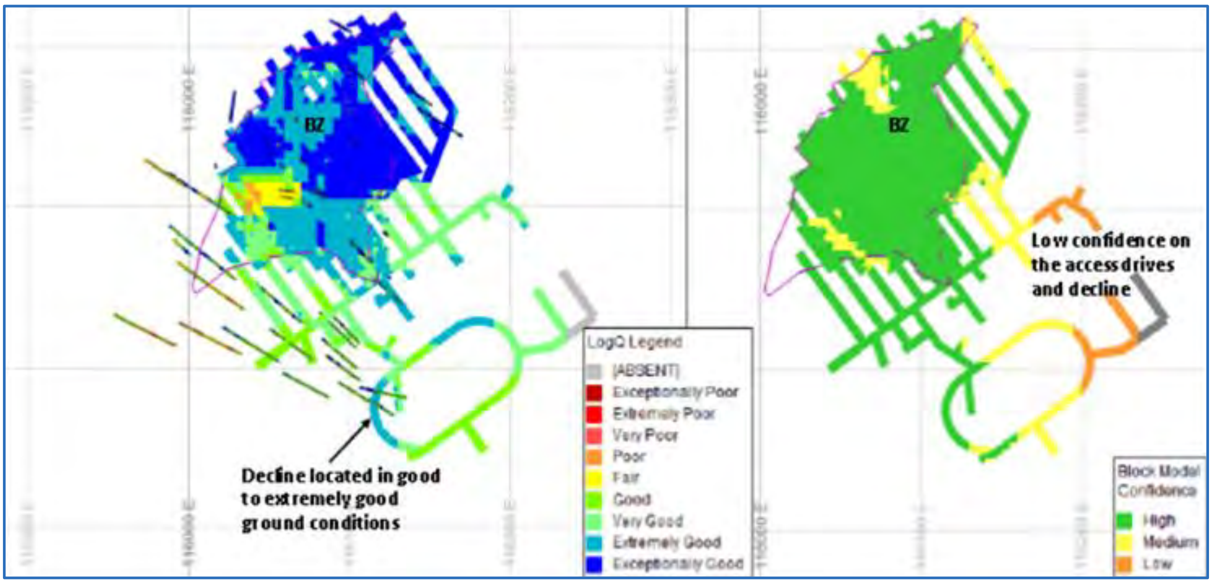
SRK, 2019

Figure 16.23 1,395 mL (Plan View): Rock Mass Quality and Block Model Confidence



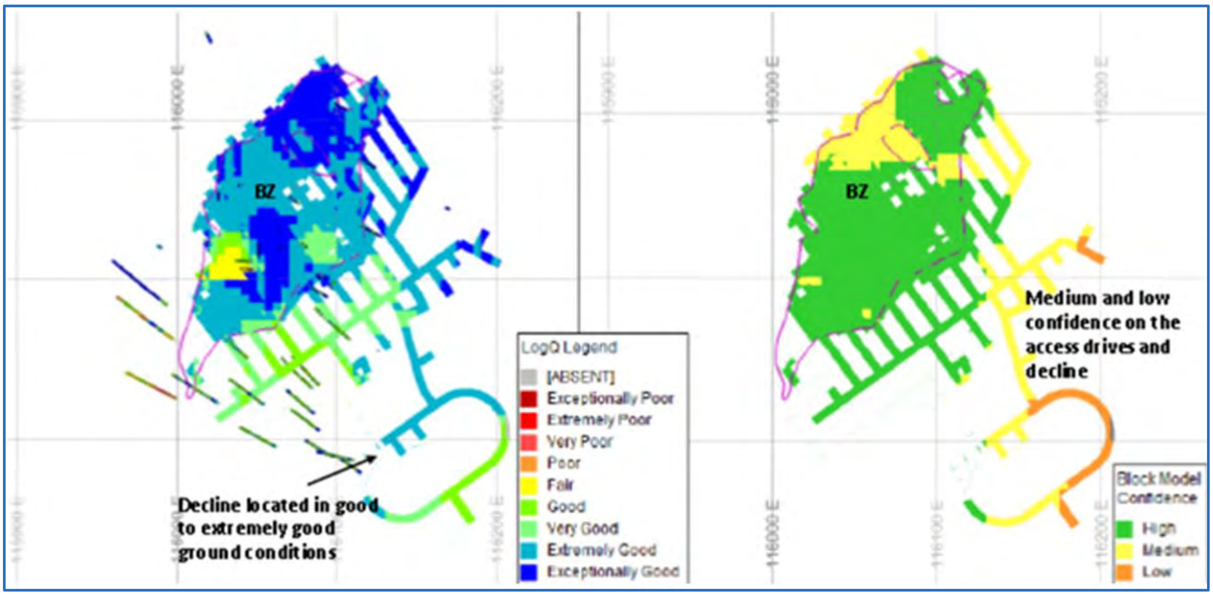
SRK, 2019

Figure 16.24 1,440 mL (Plan View): Rock Mass Quality and Block Model Confidence



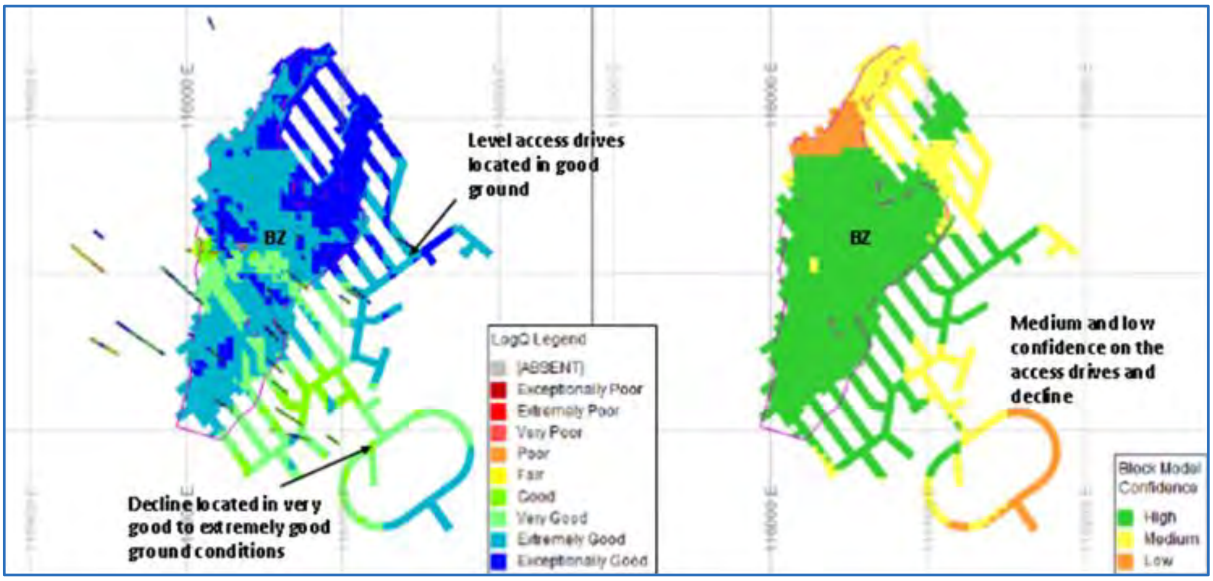
SRK, 2019

Figure 16.25 1,470 mL (Plan View): Rock Mass Quality and Block Model Confidence



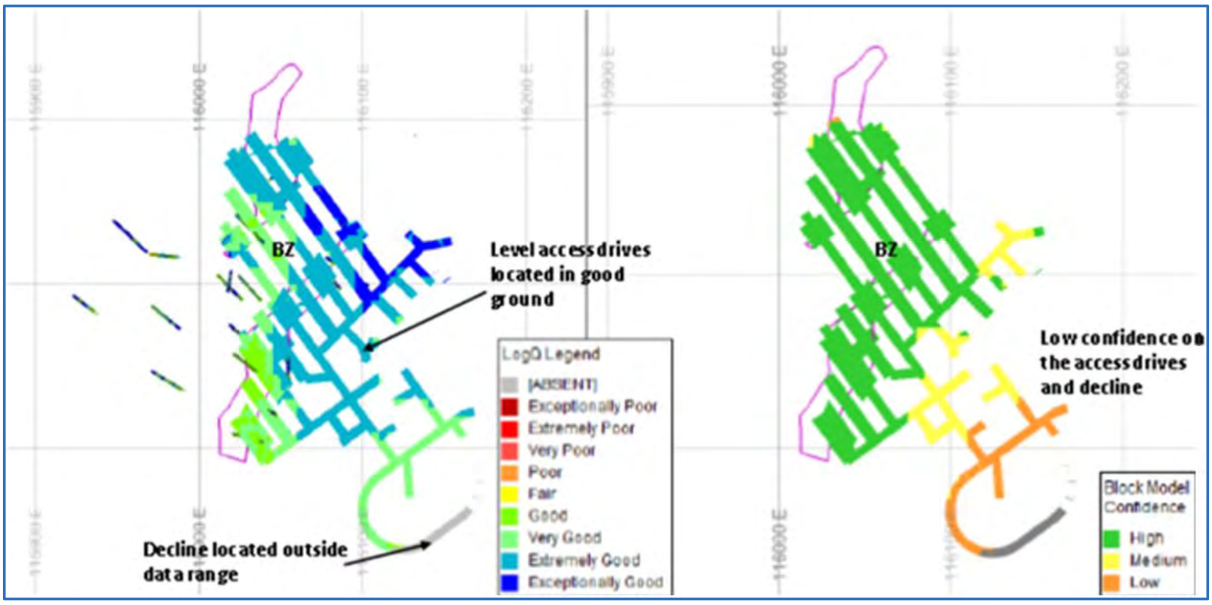
SRK, 2019

Figure 16.26 1,515 mL (Plan View): Rock Mass Quality and Block Model Confidence



SRK, 2019

Figure 16.27 1,560 mL (Plan View): Rock Mass Quality and Block Model Confidence



SRK, 2019

Support recommendations along the access levels and parallel access drives are based on the influence of stress. Where good ground conditions exist, support recommended based on the results of the elastic modelling should be applied. Where poor ground conditions exist, it is recommended that a higher support class is used, as indicated in Table 16.33.

Table 16.33 Support Recommendations for Level Access and Parallel Access Drives

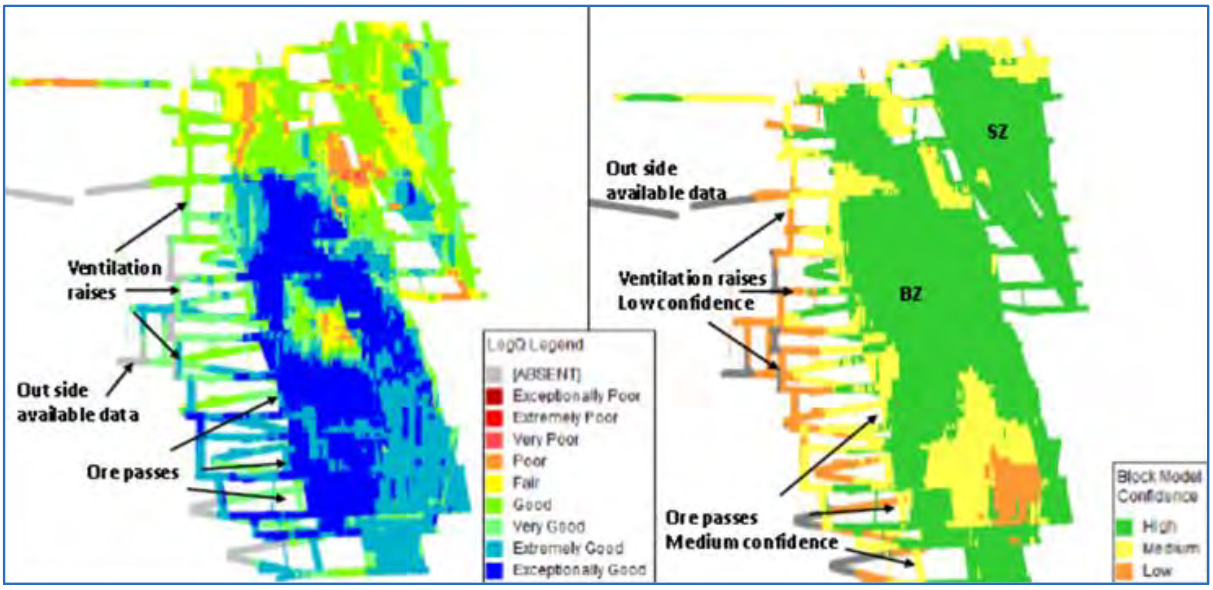
Location	Level	Recommended Support	
		Good Ground Conditions (based on elastic modelling results)	Poor Ground Conditions
Level access drives	1,220 – 1,395	S0	S1
	1,395 – 1,600	S1	S1a
Parallel access drives	1,220 – 1,515	S1	S1a
	1,515 – 1,600	S1a	S1a
Ore drives	1,200 – 1,600	S2	S3
Primary sill ore drives (BZ)	1,200 – 1,600	S3	S3
Secondary sill ore drives (BZ)	1,200 – 1,600	S2	S3
Primary sill ore drives (SZ)	1,180 – 1,395	S2	S3
Secondary sill ore drives (SZ)	1,180 – 1,395	S2	S3

Ventilation

The rock mass conditions and block model confidence for the ventilation raises are presented in Figure 16.28. In general, the ventilation raises are located either outside the block model or where the confidence in the block model is low. The confidence in the block model is high in the intake ventilation raises that are located above 1,440 mL and is medium below 1,440 mL. In general, good rock mass quality is expected in the intake ventilation raises located above 1,440 mL.

As the ventilation raises will be excavated by drill and blast utilising drop raising. It is assumed that the entry into the raises is not permitted. However, where entry into the raise is required, it is recommended that these excavations are inspected and supported with S1 support. S1 support should be used where personnel are exposed, work, and access from the supported side (top-down support sequence).

Figure 16.28 Ventilation Raises (looking South): Rock Mass Quality and Block Model Confidence



SRK, 2019

The following recommendations are made based on the block model analysis:

- The block model indicates that the decline will be developed in good to extremely good rock mass conditions. These areas will require S0 support where the rock mass rating (Q>20). There is a localised area that is in good to fair rock. In these areas, S1 support will be required (Q<20).
- The block model confidence decreases in the decline as you move further away from the stopes. There are portions of the decline that are located outside of the block model as there is no data available in these areas. The rock mass quality should be verified as the excavation progresses by ongoing mapping and inspections.
- The block model indicates that the access levels and ore drives will be in good to extremely good rock quality for the BZ. However, localised fair to poor ground conditions do occur.
- Localised areas of poorer rock mass conditions were indicated on various levels of the SZ, this may be exaggerated by the conservative logging undertaken in this area. Rock mass conditions should be verified by continuous mapping and inspections during construction.
- Where good ground conditions exist, support recommended based on the results of the elastic modelling should be applied. Where poor ground conditions exist, it is recommended that a higher support class is used.
- In general, the block model confidence is lowest on the excavations located on the eastern portion of the BZ orebody. Confidence in the block model is generally high within the SZ orebody.

- The ventilation raises are located either outside the block model or where the confidence is low. The confidence in the block model is high for the intake ventilation raises that are located above 1,440 mL and is medium below 1,440 mL. In general, good rock mass quality is expected in the intake ventilation raises located above 1,440 mL. As the ventilation raises will be excavated by drop raising drilling and blasting method no support is required except when sited in poor ground conditions or poor blasting practices. Where support is required, top-down support sequence should be implemented and access through the supported side (top).

16.1.7.3 Development Assessment Based on Elastic Modelling

The assessment of the development was carried out with the use of elastic modelling (Map3D computer software), whereby, the extraction sequence used was provided by Ivanhoe and was carried out in six monthly steps. The elastic properties used in the model were obtained from the laboratory test results and rock mass classification. The potential damage on the main access levels, access ramps, and stope access levels were assessed by calculating the DF (Martin, et al. (1999) and Cai and Kaiser, 2014) throughout each phase of the project as follows:

- At the end of the scoping study to assess the stress influence on the sub-level caving (SLC) and sub-level open stoping (SLOS).
- At the beginning and at the end of the PFS, to provide support recommendations and to ensure that the layout design met geotechnical requirements.
- At the beginning and end of the FS, to confirm support requirements and ensure that the layout design took into consideration previous recommendations made, the new stope orientations and the inclusion of the SZ orebody.

The following recommendations are made based on the block model analysis:

- The influence of stress on the decline will be negligible. Stress analysis indicates that the decline will initially require S0 support where there is no stress damage (DF = 0) from 1,220 mL to 1440 mL and S1 support on the lower levels. Where the decline intersects localised geological structures L1 support should be installed to manage poor ground conditions.
- Stress influence on the level and parallel access drive will be minimal. Level access drives will require S0 support up to 1,365 mL and S1 support from 1,395 mL to 1,600 mL. The parallel access drives are located approximately 20 m from the stopes edge and S1 support will be required up to 1,440 mL. 1,515 mL to 1,600 mL will require S1a support.
- The BZ primary and secondary ore drives will experience stress damage (DF >1.0) as these tunnels are used to access the blasted stopes and will require S2 support. The ore drives will be developed only when they are needed. The secondary ore drives will be developed in the relaxed damaged ground and S2 support will be adequate to maintain stability.
- The SZ primary and secondary ore drives will be developed simultaneously for the transverse stopes. This is because the SZ orebody is narrow and S2 support will be adequate to maintain stability. The longitudinal ore drive will be developed along strike parallel to joint set 3 and S2 support will be required to maintain stability.

- Elastic modelling shows that ore drives for the BZ and SZ will require S3 support where poor ground conditions are encountered.
- The BZ sill ore drives will experience stress (DF >1.5 m) and S3 support will be required for the primary drives. The secondary sill pillars will not be extracted and will remain as permanent pillars S2 support will be required.
- The SZ sill level on 1,320 mL ore drives will be developed along strike and will experience moderate stress damage (DF > 0.5) requiring S2 support. 1,180 mL sill ore drives will be developed across strike and will require S2 support.
- Intersection support should be installed at all intersections.
- Stope brow support should be preinstalled when required at the brow position post stope blasting.
- The ventilation raises are located in the footwall approximately 30 m from the stope edge and will be developed by conventional drill and blast method. The DF will be up to 0.1 m and is considered negligible to influence the function of the raise and no support is required except when sited in poor ground condition or poor blasting practice. Where support is required, top-down support sequence should be implemented and access through the supported side (top).

The intake ventilation raises are located in the footwall approximately 30 m from the stope and will not be influenced by stress except where the orepass is located 13 m from the stope edge with DF equal to 0.15 m. This stope will be the last to be extracted and stress damage is negligible to influence the stability of the raise and no support required except when sited in poor ground condition or poor blasting practice.

16.1.8 Geotechnical Design Parameters

Geotechnical design parameters have been derived based on the geotechnical properties determined and discussed with the mine personnel and the elastic numerical modelling analysis conducted on the final mine design. Geotechnical parameters have been outlined for the stope design and the mine access design and include backfill and support requirements.

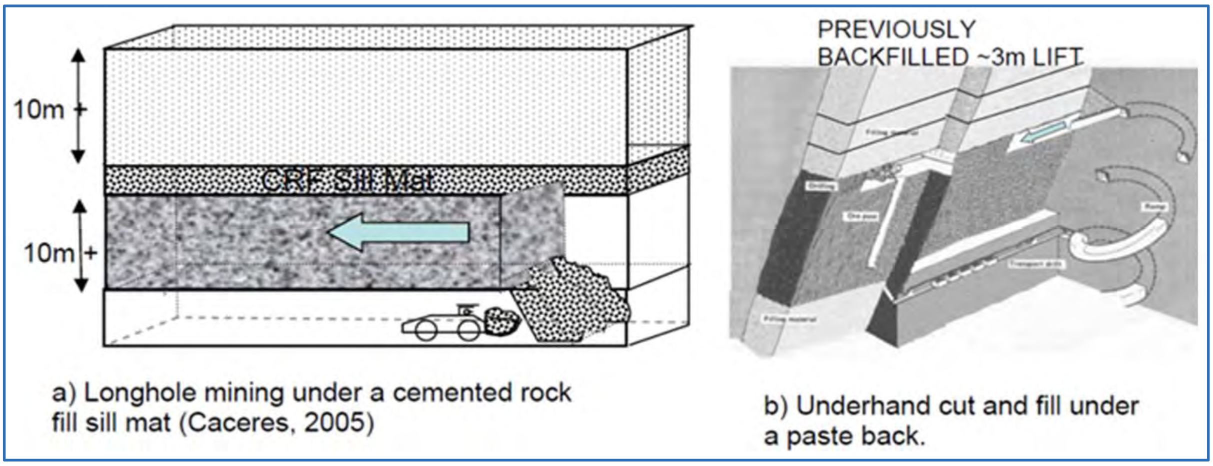
In general, ground conditions in the project area are of a good quality and at this stage no major geological structures, which could adversely affect stability have been identified. Much of the existing infrastructure and access development is safe without any support. There are minor geological structures such as zones of closely spaced joints, which could locally influence stability. Overall, the rock is very strong and very little stress damage has been observed underground and these include old production excavations on 1,220 mL.

There are areas where poor ground conditions occur and these should be identified and inspected regularly to confirm that the current support is appropriate and in reasonable condition. New support should be installed where required. Also, with increasing depth and the influence of stope abutments, stress damage can be anticipated in future. The rock is strong and brittle, therefore, rock bursts are likely to occur when the anticipated stress damage is significant.

Some of the existing major infrastructure is sited in laminated rock conditions in the footwall (located in the upper Kakontwe dolomite/Série Récurrente). These areas fall outside of the logged data, however, observations made from underground visits carried out in May 2014, October 2016, November 2017, and August 2018 indicate that the rock mass conditions in these areas are fair to poor and is thus of a lower quality compared with the good rock mass conditions identified within the vicinity of the BZ orebody.

The total mining height of longhole stopes is 60 m (comprising of an upper 30 m high stope and lower 30 m high stope) which will be separated by 15 m high sill pillars. The longholes stopes will be mined with a bottom-up mining sequence whereby the lower stope is extracted first followed by the upper stope. The stopes will be extracted using a primary and secondary longhole stoping sequence with post filling (see Figure 16.29). Only the primary sill pillars will be extracted. The sill pillars are likely to experience higher stress conditions and will require more geotechnical management. Mining is planned between 1,200 mL and 1,600 mL for the BZ and between 1,180 mL and 1,395 mL for the SZ.

Figure 16.29 Example of Undercutting Backfill



Pakalnis, 2005

16.1.8.1 Longhole Stopes with Post Backfilling Longhole Stopes

Geotechnical recommendations for longhole stopes are as follows:

- BZ stope parameters:
  - Cross-sectional dimensions will be 15 m wide and 30 m high.
  - Maximum unfilled stope lengths will be 60 m to limit the HR of 6.0 m for the stope back.
  - Stope back and wall support will not be required, providing the maximum dimensions are not exceeded.
  - The FS stope orientation will be 144°.
- SZ stope parameters:
  - Transverse cross-sectional dimensions will be 15 m wide and 30 m high.

  - Maximum unfilled transverse stope length is limited to 30 m.
  - The transverse stope orientation will be 22°.
  - The longitudinal stope orientation will be 112°.
  - Longitudinal cross-sectional dimensions will be 20 m high and width will vary according to the orebody thickness.
  - Maximum unfilled longitudinal stope length is limited to 15 m.
  - Stope back and wall support will not be required, providing the maximum dimensions are not exceeded.
  - The SZ lower sill level stopes (10 m W x 15 m H) will be extracted using the longitudinal sublevel open stope method. The upper sill pillar level stopes (10 m W x 20 m H) will be orientated in the transverse direction.
- No entry is permitted in the stopes, hence broken ore should be removed using remote Load Haul and Dump equipment (LHDs).
- Stopes should not be left open for long periods as rock conditions will deteriorate.
- Adjacent secondary stopes should not be mined simultaneously and backfill should be placed and allowed to cure prior to mining the adjacent secondary stopes.
- Due to the high stresses expected and the process of undercutting the backfill, stope/ore loss is likely during the extraction of the sill pillar stopes.
- Lower production rates for the extraction of sill pillar must be applied, due to challenges that may be faced and consider the time required to install additional support (Section 16.1.7.1) and the placement of backfill.

Backfill

The following geotechnical recommendations apply for the placement of backfill in the stopes:

- The Kipushi stopes will be backfilled with crushed rock fill containing binder.
- The BZ primary upper stopes will require 400 kPa backfill strength to achieve free standing height.
- The BZ secondary lower and upper stopes sidewall will not be exposed, however, the backfill face will require 220 kPa strength to achieve free standing height to avoid backfill collapse into the broken ore for the stopes that are longer than 60 m. The secondary lower and upper stopes that have a total length less or equal to 60 m will not require free standing strength requirements as the stope face will not be exposed.
- The primary lower stopes will require 1.2 Mpa backfill strength to take into consideration undercutting.
- The placement of backfill in the lower primary stopes using trucks must be well controlled to avoid the formation of cold joints along the natural angle of repose. The risk associated with these cold joints is that layering will be created in the backfill, which could reduce the sill strength and results in backfill sill collapse and sill pillar stope losses.

- The SZ primary transverse stopes will require 310 kPa backfill strength to achieve free standing strength requirements.
- The SZ longitudinal stope face will be exposed and will require 220 kPa backfill strength to achieve free standing strength requirements.
- There will be no undercutting of the backfill in the SZ stopes and hence no backfill sill strengths required.
- A method of tight filling will be required to maximise the recovery of the secondary stopes.
- Detailed investigations will be required to determine the effect of the fill rock minerals on the binder and long term backfill strength.
- The strength and effect of the cold joints of the poured backfill will need to be tested during the execution stage of the project.
- The proposed waste rock barricades are suitable for the containment of CRF containing approximately 92% (wt) solids.

16.1.8.2 Mine Access

Access to the stopes will be provided by utilising the existing and planned shafts, decline, access drives, and stopes drives. Exhaust ventilation raises will be developed adjacent to the stopes. The ventilation intake for the orebody will be provided by the ramp, Shafts P1 and P5.

The existing decline will be utilised from 1,330 mL and will be developed further in the footwall to a depth of 1,600 m to connect the mining sub-levels and ventilation raises. The sub-level access drives are separated by vertical distances of 75 m. The lower access drives have a vertical spacing of 30 m (based on a 30 m high stope). The secondary sill pillar drives are 6 m H x 10 m W to maximise recovery since the sill stopes will remain as permanent pillars.

Decline, Shafts, Large Excavations, and Ventilation Raises

The following geotechnical recommendations apply to the shafts, declines, large excavations, and ventilation raises:

- The existing shafts, declines, and excavations planned for use must be inspected and re-supported appropriately where required.
- Large and important excavations should be developed in good quality rock (Q>10) (87% based on current data) at least 20 m from stope excavations to avoid stress damage.
- The decline will require S0 support from the 1,200 mL to 1,440 mL (low stress environment) and S1 support from 1,440 mL to 1,600 mL (Table 16.29, Table 16.30 and Table 16.31). S1 support will also be required above 1,440 mL, where Q<20.
- Where major adverse geological structures occur, the excavation should be developed at a large angle (>45°) to the strike of the structure. Note that no major adverse geological structures have been identified at this stage.

- Ventilation raises will experience minimal stress damage with no support required except when sited in poor ground conditions or due to poor blasting practices. Where support is required, a top-down support sequence should be implemented, and access should only be allowed through the supported portion of the shaft to ensure safety of personnel.
- All tunnels must have an arched profile.

Stope Access Development

The geotechnical recommendations apply to stope access development:

- Level and parallel access drives should not be located less than 15 m away from the nearest stope to avoid stress interaction.
- Primary and secondary stope drives will generally experience higher stresses than the access drives and rockbursts may occur. They will, therefore, require S2 support.
- Primary sill pillar stope drives will require S3 support as these are in a high stress environment.
- S2 support should be installed in secondary sill pillar stope drives (this is since there will be no re-entry into these stope drives).
- Brow support should be pre-installed when required at the stope brow position, post stope blasting.
- Where secondary stope drives are developed after the extraction of the primary stopes, S2 support will be required.
- All tunnels must have an arched profile.

16.1.8.3 Monitoring

A 3D in-mine seismic monitoring system will be required to accurately gauge the rock mass response to mining. Two seismic system suppliers: IMS and ESG, were contacted by SRK to provide quotations for an underground seismic network:

- The IMS seismic system design is based on 21 underground sites with roughly 12 geophones in the decline area and eight geophones in the P5 shaft and link drive areas, supplemented by one or two surface sites. Each sensor site will require a 15–50 m borehole drilled with a downward dip between a few degrees and vertical to give as much lateral spread to the array as possible. The design is based on IMS' experience and judgment to design an appropriate sensor array for the monitoring regions. IMS have also run a sensitivity analysis for the planned 21 sites.
- ESG provided quotes for two options of system sensitivity that are based on their experience with the existing ESG system installed at Glencore's Nickel Rim South (NRS) Mine. This comparison is relevant as the volume of monitoring interest at the Kipushi Mine is comparable to NRS, and rock mass conditions are believed to be similar. The average sensor spacing at NRS in May 2016 was approximately 120 m (one sensor per 120 m^3). The Gutenberg-Richter relation for NRS indicates that this density of sensors results in complete system sensitivity down to moment magnitude (Mw) –2.0 within the volume of interest, with some areas of the mine being covered below this magnitude threshold (Mw –2.8). It is known that seismic event location errors at NRS are typically less than 10 m.

In addition to seismic monitoring, the following will also be required:

- Closure monitoring with the use of closure metres/extensometers will be required to measure convergence (rock displacement) of the stope drives and vulnerable access drives. This will be extremely important during the extraction of the sill pillar.
- Elastic modelling of mined and planned stopes, as well as development tunnels, should be carried out on a regular basis to determine where high stress concentrations are located. This will allow for updates to the planned design or for increased support to be applied in the affected areas.
- Stope Assessments should be conducted on a regular basis by the appropriate personnel (strata control officers, rock engineers etc.) where the rock mass conditions, and the stability of the stopes are observed.
- Stope Reconciliation should be conducted based on assessments of dilution and the application of laser cavity monitoring scans. This will allow for an improved understanding of stope behaviour.
- Ongoing geotechnical mapping should take place at regular intervals in the developments to verify the rock mass conditions determined in the FS. This will also allow for the identification of localised weak zones and potentially unstable wedges, which should be supported as outlined in Section 16.1.7.1.

16.1.9 Geotechnical Recommendations

The following additional geotechnical investigations are recommended for the construction phase of the project:

- The review of the stress field indicates that the horizontal stress magnitudes are different, and this highlights the need to carry out stress measurements during the execution stage of the project.
- The performance of CRF in stopes will need to be investigated through trials in the early stopes. It is important to confirm that the design strengths, avoidance of cold joints, tight filling, and effective waste rock barricades can be achieved. The trials should include drilling into the placed fill and collection of samples for inspection and testing. The first sill pillar stopes must be mined carefully and monitored to assess the stability of the backfill being undercut.
- Drill geotechnical holes to determine ground conditions at each ventilation raise.

16.2 Mine Design

Mining zones included in the current Kipushi Mine plans occur at depths ranging from approximately 1,200 mL and 1,590 mL with 0 mL being the surface. Access to the mine will be via multiple vertical existing shafts and internal decline. Mining will be performed using highly productive mechanised methods and CRF utilised or backfilling of open stopes. Depending on the required composition and available material, excess waste rock, and dense media separation (DMS) tailings will be used in the CRF mix as required.

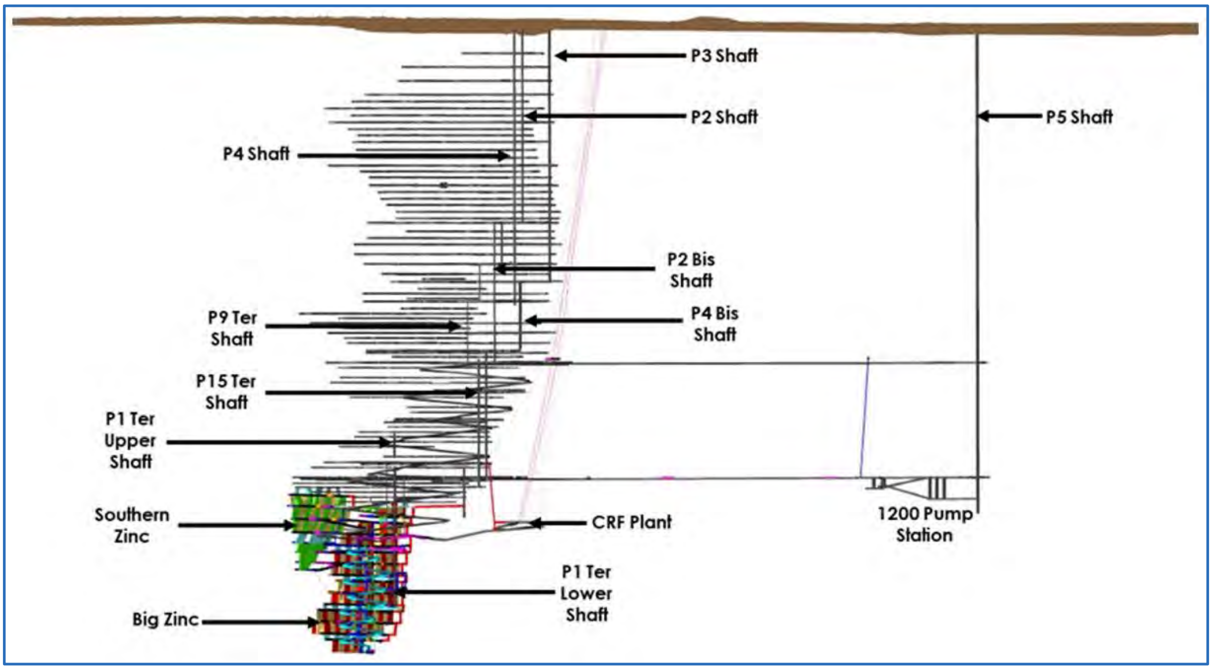
The Mineral Reserve estimate for Kipushi was based on the 14 June 2018 Mineral Resource reported in the Kipushi 2019 Resource Update. Only Measured Resources have been used for determination of the Proven Mineral Reserve and only Indicated Resources have been used for determination of the Probable Mineral Reserve.

All pertinent technical and economic data related to the mining of the resource was provided by Ivanhoe. All dollar amounts throughout the report are expressed in US Dollars ($).

16.2.1 Current Mine Layout

The existing mining infrastructure consists of five surface vertical shafts and a number of sub-vertical shafts allowing access to deeper levels (Refer Figure 16.30).

Figure 16.30 Kipushi – Planned and Existing Development



OreWin, 2022

The 850 mL will be utilised as intermediate level on the Shaft 5 to allow personnel and equipment to enter the mine workings without doing so via the main haulage and crusher level, minimising interactions, and downtime to the haulage network.

The main working area is connected to Shaft 5 via the 1,150 mL main haulage level. There is a crusher chamber at 1,200 mL; the crusher level is now dewatered. The underground infrastructure exposed since dewatering is in relatively good order. The crusher is being replaced as the cost of refurbishment was determined to exceed the replacement cost.

A 5 m H x 5.8 m W decline was developed from 725 mL to approximately 1,330 mL, the upper to deeper working levels and the top of the Big Zinc.

A network of underground pumps, cascading dams and pipework currently dewaters the mine at a maximum rate of 3,500 m^3/h.

Workshops and magazines exist on the 1,132 mL and 850 mL. These areas require rehabilitation but will provide locations for machine maintenance, breakdown areas, welding bays, wash bays, tyre changing and storage, explosives storage, lubricant tanks, and diesel storage.

Mine Access will be via the existing shafts and internal decline to Big Zinc. The decline will be extended from the current position. Mined material will be trucked to the 1,150 mL drive crusher tip, fed to the crusher on the 1,200 mL and then conveyed to silos for temporary storage before being hoisted to the surface up Shaft 5.

Fuel is supplied via a 1,325 m long fuel line, from the surface to the 850 mL workshop and from the 850 mL workshop to the 1,332 mL workshop. Fuel will be piped to hose reel stations for underground equipment refuelling. The fuelling station will have the storage tanks and pumps installed in an enclosed drift with fire doors and appropriate fire suppression systems.

The equipment requirements for the Kipushi Project are split into two categories, fixed equipment, and mobile equipment. The equipment requirements for each category cover the major components for the operation.

The mobile equipment required for lateral development includes drill jumbos, LHDs, haul trucks, and ground support equipment. Mobile equipment required for stoping includes longhole drill rigs, LHDs, haul trucks, and ground support equipment.

Due to the historic nature of Kipushi and the fact it is currently under care and maintenance, significant underground fixed equipment exists in place. This includes shaft winders, skips, and cages, workshop facilities, silos, conveyors, and dewatering pumping infrastructure.

The existing crusher chamber and accompanying excavations on the 1,150 mL at Kipushi are currently being rehabilitated and will be recommissioned. The existing Crushing and Ore handling infrastructure will be replaced.

The site personnel are provided partially by the client and partially by the contractor. Both provide a combination of expatriates and nationals. The expatriates are employed at the beginning of the project, to be replaced by nationals as the project advances. The client provides labour for roles from the surface down to and including the crusher, while the contractor provides labour from the crusher down to the face.

16.2.2 Mining Method

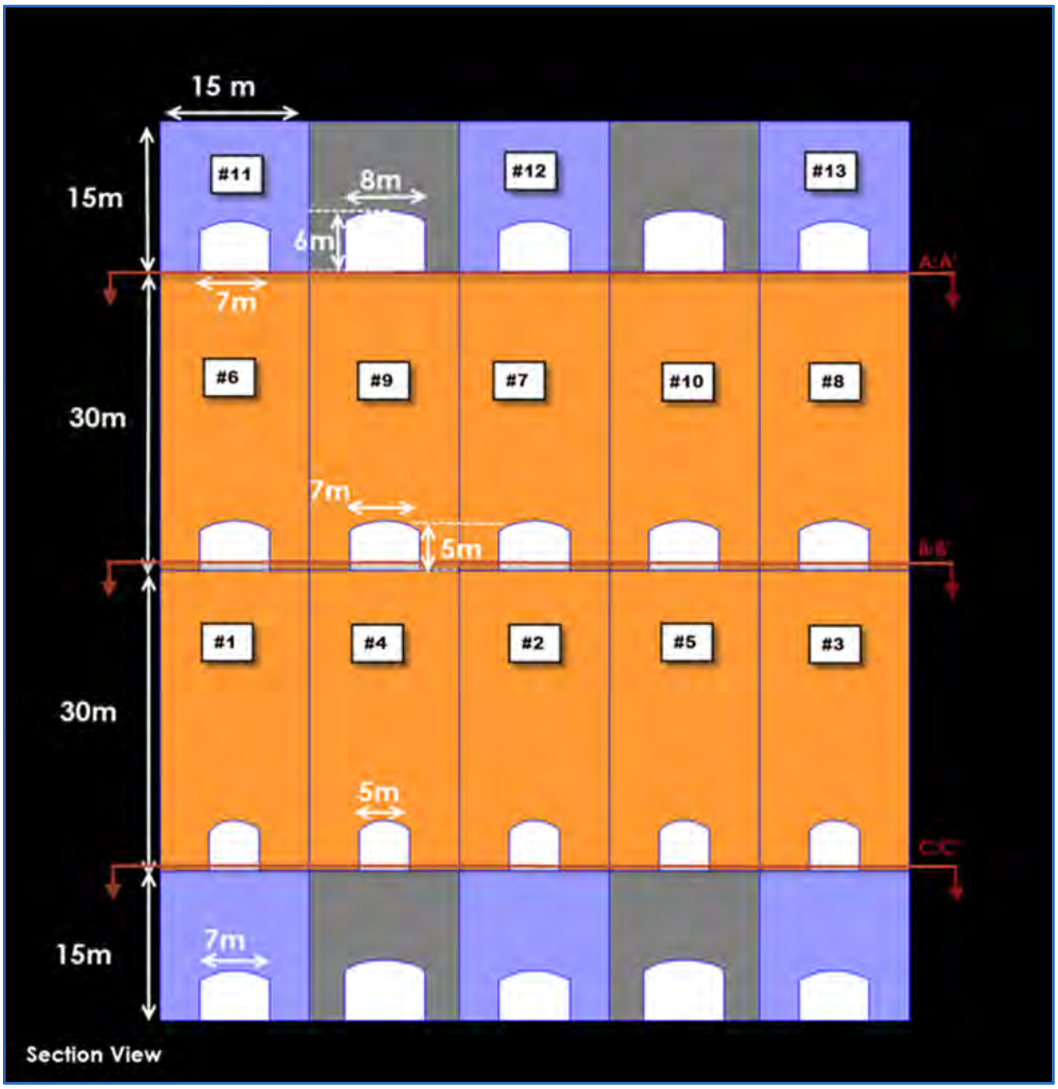
Access will be via the existing shafts and internal decline to Big Zinc. The decline will be extended from the current position. Mined material will be trucked to the 1,150 mL drive crusher tip, fed to the crusher on the 1,200 mL and then conveyed to silos for temporary storage, before being hoisted to the surface via Shaft 5.

The Kipushi PFS assessed a number of mining methods, and the results of this work were used as an input to the FS. The PFS mining method selected was largely carried through into the FS, with some variations. Geotechnical input validated an alternate orientation of stopes, based on analysis of the major joint structures in the Big Zinc area. Stope layouts were also revised from longitudinal to transverse, which resulted in a reduction of development waste required to set up the number of panels required to be in production concurrently. In consultation with BBE Consultants, an improved primary ventilation layout was also achieved which provided improved distribution of chilled, fresh air directly to the work areas, coupled with efficient exhausting from each level.

Mining is planned to be a combination of Transverse Sublevel Open Stoping and Pillar Retreat mining methods. The Big Zinc area stopes are planned to be mined as Sublevel Open Stopes to be extracted in a Primary and Secondary sequence and filled with CRF. The sill pillars are to be mined on retreat once the stopes below and above have been mined.

Figure 16.31 depicts a schematic of the transverse stoping layout. The primary and secondary stopes are each 15 m W x 30 m H, with panels of two sublevels separated by a 15 m high sill pillar every 75 m vertically. Stope lengths vary from 5–60 m (maximum). Ore drives are developed perpendicular to the strike of the orebody, at a spacing of 15 m centre to centre. Drill drive dimensions are 6.0 m W x 6.0 m H and extraction drives are 5.5 m W x 5.0 m H, both with an arched profile.

Figure 16.31 Big Zinc Stope Cross-Section

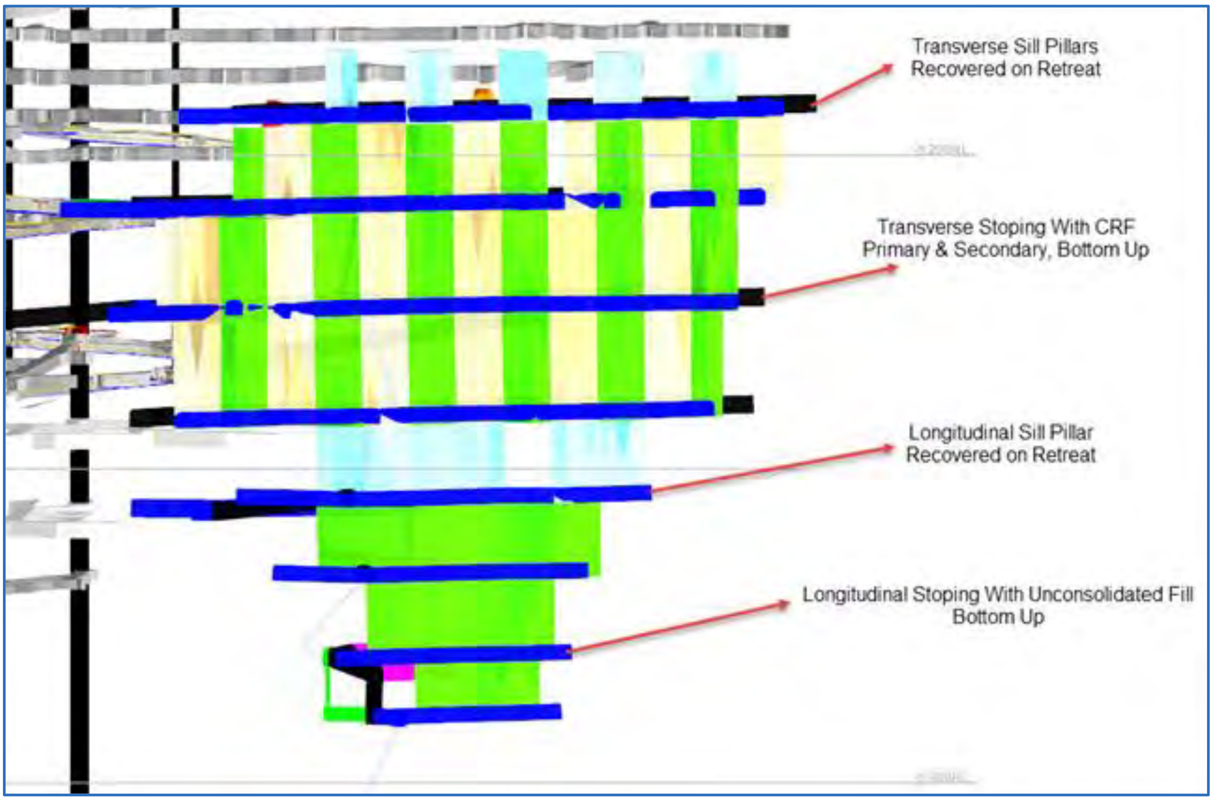


OreWin, 2017

On the sill pillar levels stopes are 15 m W x 15 m H, with stope lengths varying from 5–60 m. Drill drive dimensions are 6.0 m W x 6.0 m H, with drill drives located in permanent pillars designed at 8.0 m W x 6.0 m H, to maximise ore extraction.

The Southern Zinc area is predominantly planned to be extracted by the same Transverse Sublevel Open Stope method. Additionally, a thinner area at the base of the Southern Zinc zone is planned to be mined by longitudinal open stoping on retreat, bottom-up with unconsolidated fill. Figure 16.32 shows the Southern Zinc transverse method in green/yellow and longitudinal retreat area at the bottom of the figure.

Figure 16.32 Southern Zinc Mining Method



Mining Plus, 2019

16.2.3 Economic and Processing Assumptions

For the purposes of mine planning (designing the stopes and determining development material classification as ore or waste) Net Smelter Return (NSR) and cost calculations were used, defined as NSR10 values. NSR is the net revenue received from the sale of the metal product minus refining and transportation costs. The value used for the cut-off of the project is not the breakeven cut-off but was elevated to cover capital costs and profit margins. As the orebody is not diffuse the high grade is relatively consistent within the confines of the mineralisation, with limited lower grade material on the peripheries, such that the elevated cut-off value did not impact inventory too greatly.

The supplied Resource Model included NSR10 values, and additionally updated NSR values based on updated parameters which where coded into the block model as attribute NSR13. Because of the high value of the mineralised zone relative to the mining cost, the mine design is not overly sensitive to the accuracy of these calculations. NSR10 was used as the main grade attribute for targeting high grade material as in the schedule. The parameters used in the calculation of NSR10 values are shown in the model, NSR10 was calculated on a block by block basis using testwork algorithms, formulae, prices, recoveries, and costs shown in Table 16.34 and Table 16.35. Each block was assigned a dollar per mined tonne value NSR10.

Table 16.34 NSR10 Modifying Factors

Maximum Concentrate Zinc Grade	%	60.58
Maximum Zinc Recovery	%	95.00
Maximum Mass Pull	%	95.00
Payable Zinc Metal	%	85.00
DMS Concentrate Recovery Constants		
Testwork Zinc tail grade	%	10.76
Testwork Zinc feed grade	%	45.36
Zn non-floating	%	0.10
Concentrate Moisture Content	%	12.00
Zinc Metal Price	$/lb	1.01
Treatment Charge	$/t dmt	200.00
Concentrate Transport Cost	$/t conc.	249.61
DRC Royalty	% smelter payables	2.00
Gécamines Royalty	% smelter payables	2.50
DRC Export Tax	% value of the export	1.00

Table 16.35 Zinc Recovery and Concentrate Algorithm

Zinc Concentrate Recovery	%	$0.00000009 * Zn_{Grade}^3 - 0.000004 * Zn_{Grade}^2 + 0.0027 * Zn_{Grade} + 0.831$
Mass Pull	%	$0.017 * Zn_{Grade} - 0.0583$
Concentrate Zinc Grade	%	$\frac{Zn_{Recovery} * Zn_{Grade}}{\text{Mass Pull}}$
Tail Zinc Grade	%	$Zn_{nf} \frac{+ (Zn_{tail\ ref} - Zn_{nf})}{Zn_{ref\ Grade}} * Zn_{Grade}$

Elemental smelter penalties were not included in the NSR10 calculation.

16.2.4 Cut-off Policy – Ore/Waste Determination

The cut-off strategy is based on the NSR. NSR is calculated from each blocks grade and expected metallurgical recovery. Only material falling inside the Measured or Indicated Mineral Resource category was included in the NSR calculation. Sometimes, material which is not Measured or Indicated can be included in the design (as it cannot be avoided where it is required to make up a practical mining shape) but this is included at zero grade.

Stopes were designed to an elevated cut-off of $135/t NSR10. This policy maximises the NSR above cut-off in each stope where:

$$\text{NSR above Cut-off} = (\text{Tonnes}) \times (\text{NSR-Cut-off})$$

This elevated cut-off value is in line with the work undertaken during the PFS and previously documented in the NI 43-101 Technical Report – Kipushi 2017 Pre-feasibility Study published to System for Electronic Document Analysis and Retrieval (SEDAR) on 25 January 2018.
The cut-off is elevated above the break-even cut-off grade (Section 15) to ensure that the highest grade is targeted. At this elevated cut-off, there is a relatively sharp boundary between the ore boundary and surrounding waste. This sharp cut-off results in low sensitivity of the stope tonnes to the cut-off.

Material from development drives will be mined, regardless of its NSR grade. After the mining costs have been sunk, the incremental costs of processing it rather than using it as backfill are much lower. Because of this, a marginal cut-off of $50/t NSR10 is used to determine which development material will be treated as ore and waste. The marginal cut-off only accounts for the processing cost, as well as the opportunity cost of not using this waste for backfill.

16.2.5 Geotechnical Parameters

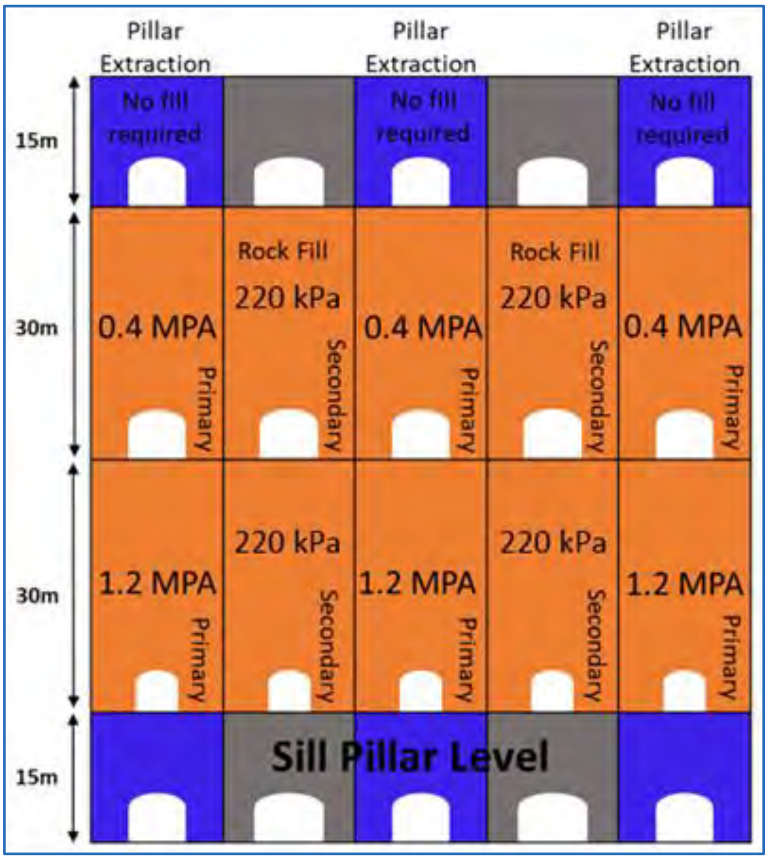
Geotechnical design parameters have been provided by SRK Consultants based on drillhole data, underground mapping, discussions held with mine personnel, and elastic numerical modelling analysis conducted on the FS mine design. Geotechnical parameters have been outlined for the mine access design and the stope design and include backfill and support requirements.

It was determined that CRF would be the most suitable method to provide backfill for the mine design.

For all stopes in the Kipushi design the backfill strength requirements (Figure 16.33) are as follows:

- Primary lower stopes 1.2 Mpa,
- Primary upper stopes 400 kPa, and
- Secondary (lower and upper) stopes 220 kPa.

Figure 16.33 Kipushi CRF Design Strengths



Mining Plus, 2019

For more detail, refer to Section 16.1.

16.2.6 Stope Dilution

The stope extraction sequence of primary, secondary then tertiary stopes largely governs which edges of any given stope may be exposed to dilution in the form of over or underbreak. The quantity and quality of dilution material are determined by understanding the exposure to fill and surrounding rock. The stope dilution factors that have been applied are shown in Table 16.36.

Table 16.36 Stope Dilution Factors

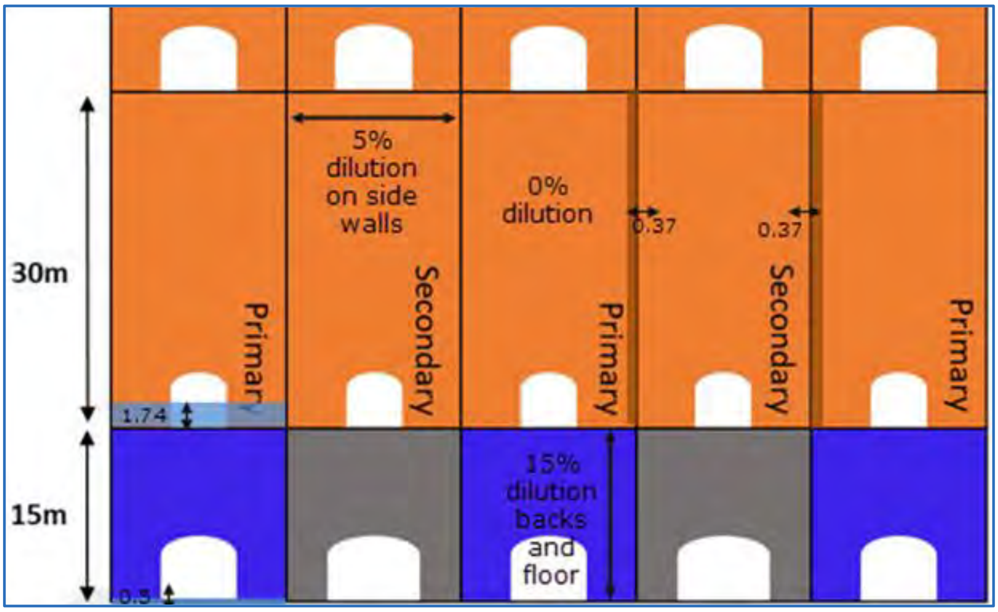
Primary	0%
Secondary	5%
Tertiary	15%
Southern Zinc	15%

Dilution for primary stopes is assumed to be zero as any dilution at this stage will be from surrounding stopes, so any overbreak is assumed to be more of a similar grade to the stope being taken. Subsequent stopes will have dilution from backfill or low-grade material at the mine boundary.

As seen in Figure 16.34 most secondary stopes will be blasted against large fill exposures in either sidewall. The blasting of these stopes is anticipated to over break into either wall by 0.37 m, representing an overall dilution of 5%.

Tertiary stopes will incur dilution in the backs and floors in a similar way to the walls of the secondary stopes. The overall dilution for tertiary stopes is estimated to be 15%.

Figure 16.34 Stope Dilution



Mining Plus, 2019

Stopes in the Southern Zinc mining area have had a dilution of 15% applied, as the stopes are generally narrower, and thus incur a higher proportion of dilution relative to in-situ width.

16.2.7 Stope Recovery

Recovery factors were applied to primary, secondary, and tertiary stopes to account for losses that may occur during the mining process. The factors that have been applied to diluted tonnes are shown in Table 16.37.

Table 16.37 Stope Recovery Factors

Primary	95%
Secondary	95%
Tertiary	90%
Southern Zinc	95%

The main sources of loss will be:

- Incomplete remote loading,
- Underbreak, or
- Stope failure.

16.2.8 Design Layout

16.2.8.1 Stopes

Stopes were created using Mineable Stope Optimiser (MSO).

Table 16.38 shows the stope geometries used as inputs to MSO at the elevated cut-off grade of \$135/t NSR10.

Table 16.38 Big Zinc – MSO Stope Parameters

Stope Type	Height (m)	Width (m)	Minimum Length (m)	Maximum Length (m)
Primary Stope	30	15	5	60
Secondary Stope	30	15	5	60
Tertiary Stope / Sill Pillar Recovery	15	15	5	60

Stope widths and heights were set based on the geotechnical guidance provided for maximum unsupported hydraulic radii (Section 16.1.6). Stope lengths were varied up the geotechnical stope maximum length of 60 m. Areas of low grade within the mining areas were excluded as internal pillars between stopes.

Stope heights and widths are controlled by the sublevel and crosscut spacing. Stope hanging walls and footwalls were angled to best follow the mineralisation. Due to the steep dip of the orebody these are typically close to vertical.

The Southern Zinc zone has similar design parameters to the Big Zinc zone, albeit the resultant stope widths are significantly less due to the narrower width of the orebody (refer Table 16.39).

Table 16.39 Southern Zinc – MSO Stope Parameters

Stope Type	Height (m)	Width (m)	Minimum Length (m)	Maximum Length (m)
Primary Stope	30	15	5	35
Secondary Stope	30	15	5	35
Tertiary Stope / Sill Pillar Recovery	20–30	5	5	60

16.2.8.2 Development

Development is designed to be suitable for a modern rubber-tyred, diesel-powered, mechanised mining fleet. Table 16.40 shows the profiles and typical dimensions of the development drives used in the design.

Table 16.40 Lateral Development Profiles

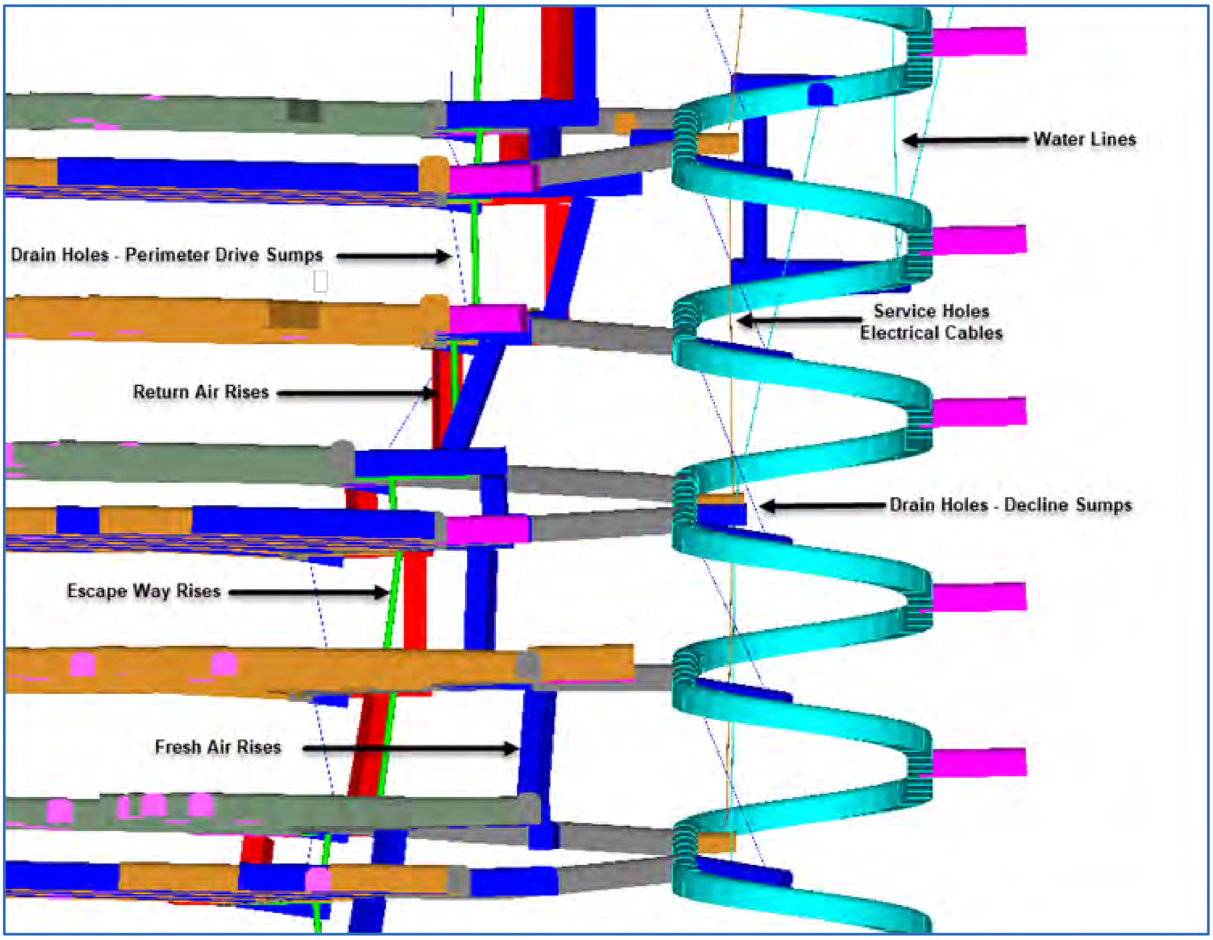
Drive Type	Height (m)	Width (m)	Gradient	Comment
Access	5.3	5.8	<1 in 7	Average gradient of 1 in 40
Backfill Stockpile	6.5	7.5	1 in 50	Additional height for truck tipping
Decline	5.3	5.8	–1 in 7	Turning radius 25 m
Drill Drive	6.0	6.0	1 in 45	Second lift
Drill Drive	8.0	6.0	1 in 45	Sill pillar drive – larger dimension to maximise ore extraction
Escapeway Drive	5.0	5.0	1 in 50	
Extraction Drive	5.5	5.0	1 in 45	
Fresh Air Way	5.3	5.8	1 in 50	
Fresh Air Way	6.0	6.0	1 in 50	
Incline	5.3	5.8	1 in 7	Turning radius 25 m
Perimeter Drive	5.3	5.8	1 in 40	
Pump Station	6.0	6.0	1 in 50	
Return Air Way	5.3	5.8	1 in 50	
Return Air Way	6.0	6.0	1 in 50	
Stockpile	6.0	6.0	1 in 50	
Substation	5.0	4.5	1 in 35	
Sump	5.0	4.7	–1 in 7	
Truck Loading Bay	6.0	6.0	1 in 50	Crusher truck tip and turning bays

Levels are arranged in panels of three lifts (Figure 16.31), linked together with a spiral decline (Figure 16.35). Based on the activities that will be completed on them, levels can be categorised into four types:

- Extraction levels
- Extraction and drill levels
- Sill Pillar / Pillar Extraction Levels
- Longitudinal Levels

Fresh air connections are located in a central position of the Big Zinc orebody. Fresh air rises connect between each level, there is a connection to the perimeter drives at each level, allowing air to enter the mine workings. Southern Zinc uses a separate fresh air network which is extended from the Big Zinc levels. This allows shorter ventilation ducting and the Southern Zinc area to be accessed concurrently with Big Zinc mining, or afterwards as schedule requirements dictate.

Figure 16.35 Spiral Decline and Vertical Service Connections Between Levels



Mining Plus, 2019

Return air drives are positioned on the north-east end of the Big Zinc perimeter drives. They are connected to by return air rises between each level, facilitating return air to exhaust from each level of the mine. The Southern Zinc mining area uses the perimeter drives of Big Zinc to exhaust, due to the position of the return air rises to the north.

The mine's egress system is formed by escape way drives which are sub-vertically joined by escapeway rises between each level. The drives are generally situated in a central position of the perimeter drives close to fresh air drives. Southern Zinc connects to the Big Zinc levels to utilise the same egresses.

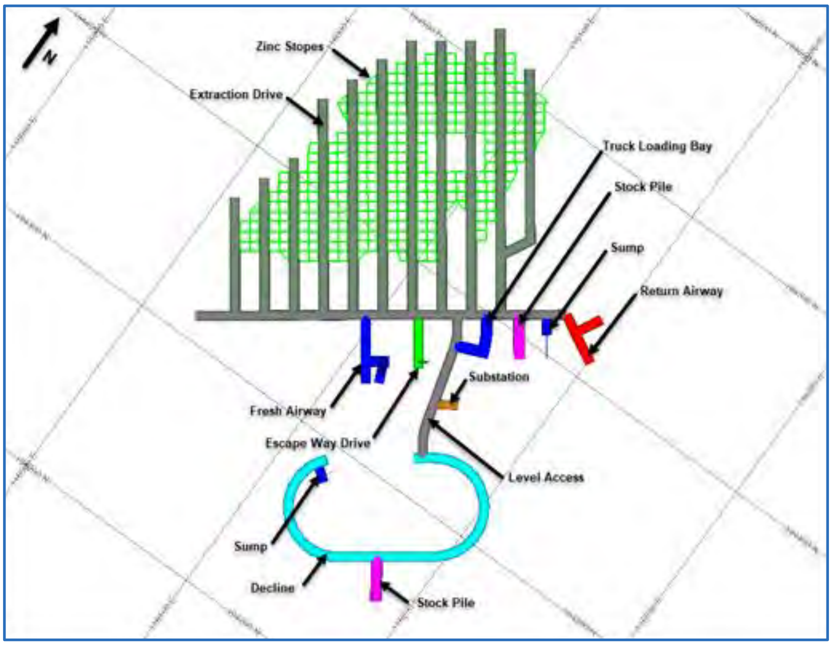
The dewatering system consists of sumps located on the level perimeter drives and decline. The sumps are connected by drain holes enabling water to flow by gravity to lower levels of the mine before they would overflow. On reaching pumping sumps or pump stations, water will be lifted in stages to the surface through pump lines and dedicated rising mains.

Electrical power is delivered to the levels by substations situated on the declines. Substations are joined vertically through service holes that allow power cables to be run to lower elevations with reduced run lengths.

The typical level consists of a perimeter drive with 10–12 ore drives spaced 15 m apart, centre-to-centre. An average ore drive length is 120 m, and the average stand-off distance from stopes to the perimeter drives is 20–30 m.

Figure 16.36 shows a typical extraction level layout, generally comprising of the following excavations: ore drives, slot drives, perimeter drives, ventilation drives (fresh and return airways), truckload bay, sump, escapeway drive, and stockpile.

Figure 16.36 Extraction Level Layout

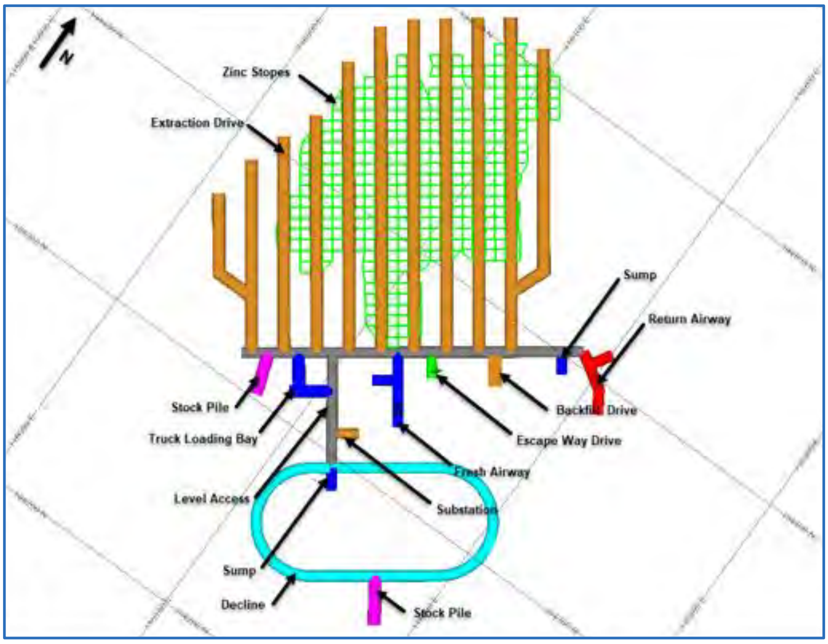


Mining Plus, 2019

Drill levels are located 30 m above extraction levels (Figure 16.37). Ore drives on these levels are used for both drilling and ore extraction. In addition to the infrastructure found on the extraction levels, backfill drives are also included, which are used during the filling process to house the backfill and/or to mix the CRF.

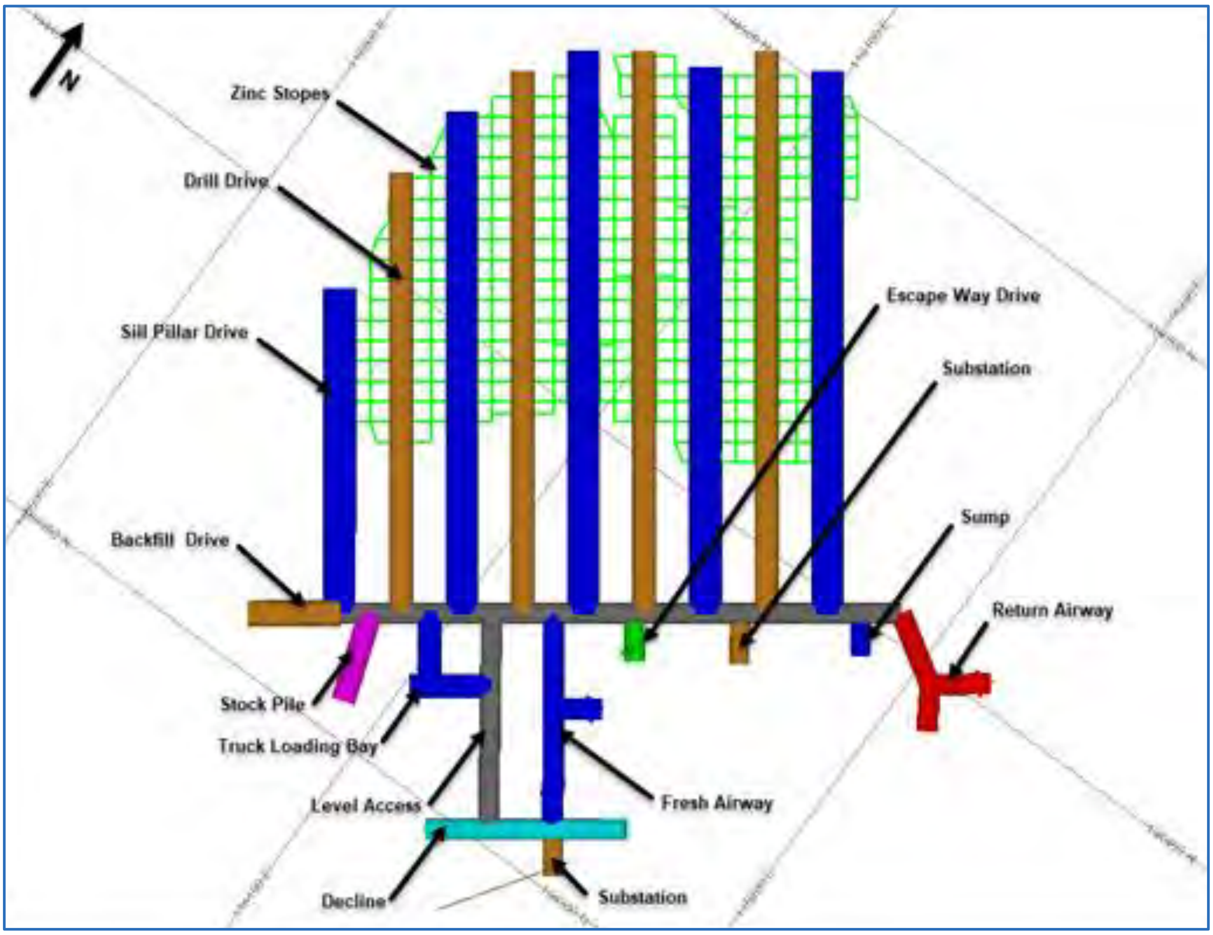
Sill pillar levels are located above extraction and drill levels (Figure 16.38). They consist of sill pillar drives, which are wider to maximise ore extraction and drill drives. On these levels, the typical excavations are perimeter drives, ore drives, ventilation drives (fresh and return airways), truckload bay, sump, escapeway drive, stockpile, and backfill drives.

Figure 16.37 Drill Level Layout



Mining Plus, 2019

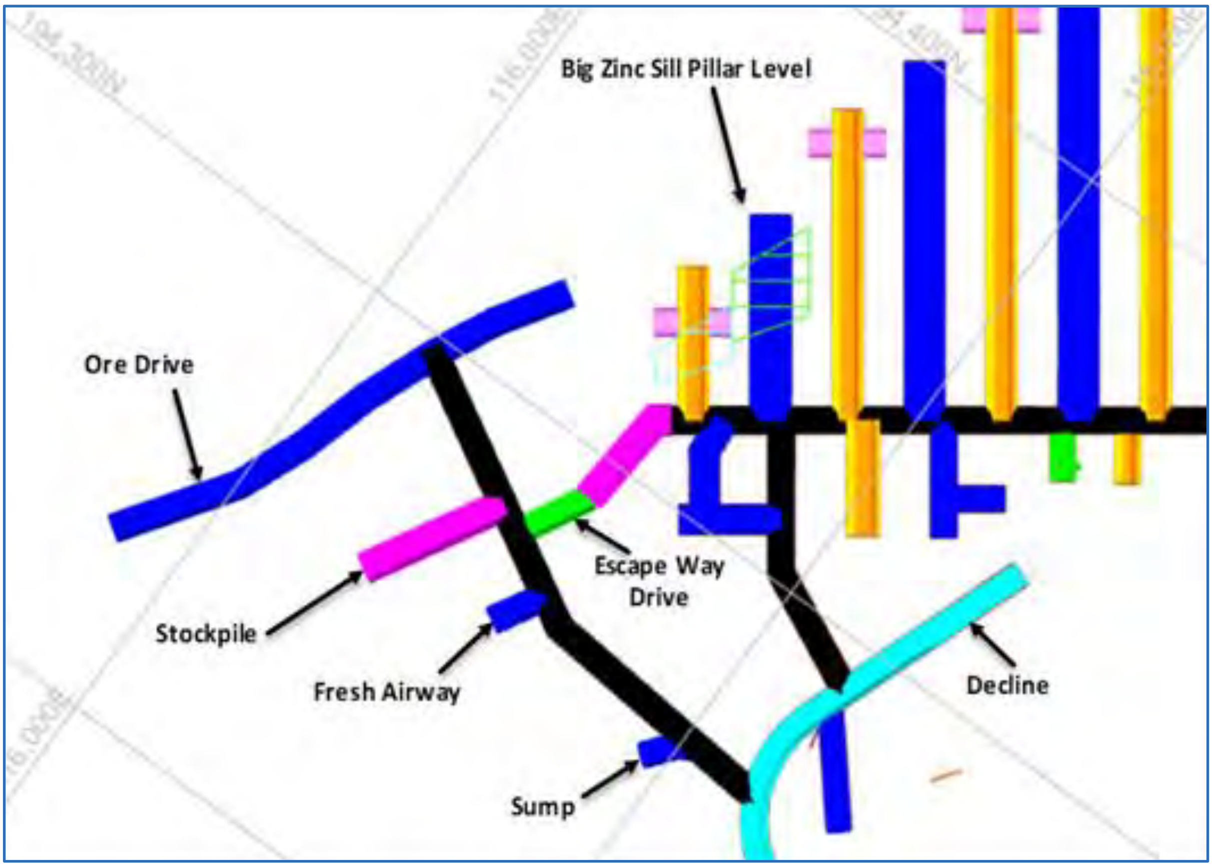
Figure 16.38 Sill Pillar Level Layout



Mining Plus, 2019

The longitudinal levels in Southern Zinc consist of an ore drive, stockpile, ventilation drive, and escapeway as shown in Figure 16.39. Due to the smaller size of the orebody, the infrastructure on these levels is reduced to the very minimum. The development will serve multiple purposes such as the stockpile which will be used for loading and backfilling instead of having a separate stockpile for each. The tonnages in this area does not justify the amount of development typically seen in the transverse levels.

Figure 16.39 Southern Zinc Longitudinal Open Stoping Level Layout



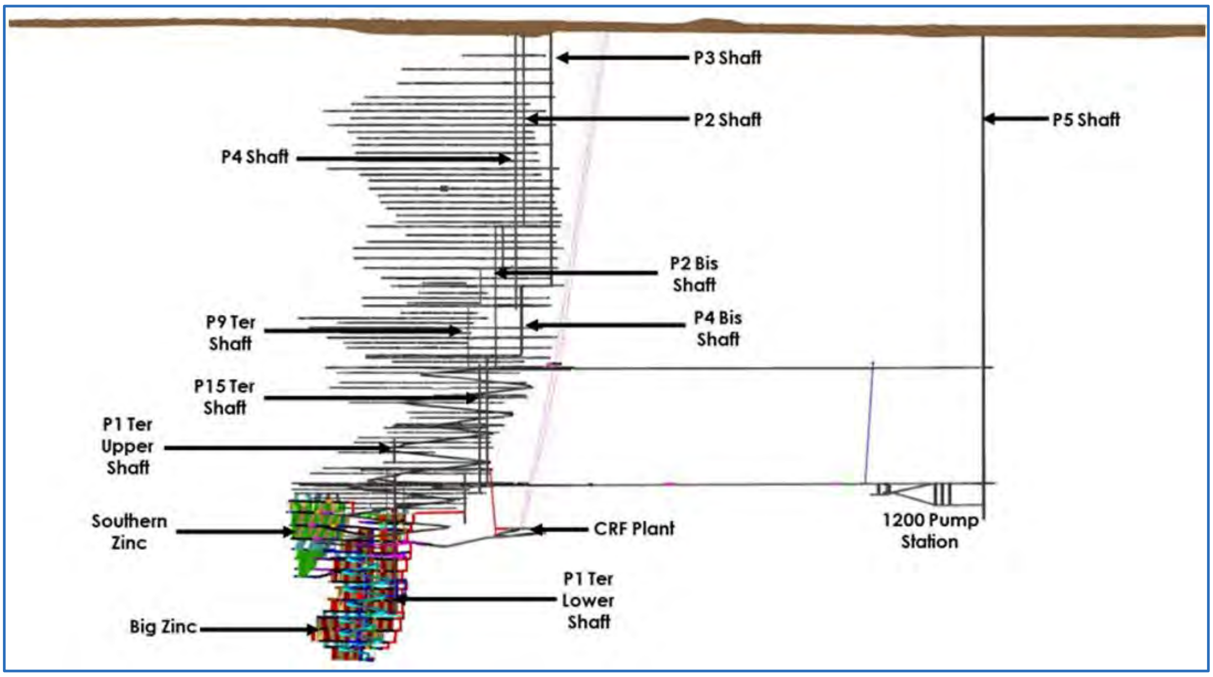
Mining Plus, 2019

16.2.9 Access

16.2.9.1 Existing

To access the Big Zinc and Southern Zinc mining areas, existing excavations will be utilised and extended. Figure 16.40 shows the location of the FS design areas relative to the existing excavations. Provisions have been made for stripping and rehabilitation of existing excavations, where survey data has identified that existing profiles are of insufficient size for the future mining operations.

Figure 16.40 Access to Big Zinc and Southern Zinc Mining Areas



OreWin, 2022

The lowest existing lateral excavation is the decline face at approximately 1,330 mL, and the lowest existing vertical excavation is the base of P1 Ter Shaft, at approximately 1,486 mL. Neither of these locations have yet been re-surveyed as both are still flooded, and further remediation work may still be required beyond what has been included in the FS.

The primary access to the mine will be via the P5 shaft, which has been refurbished and is fit for purpose for the FS design. The shaft will be used for:

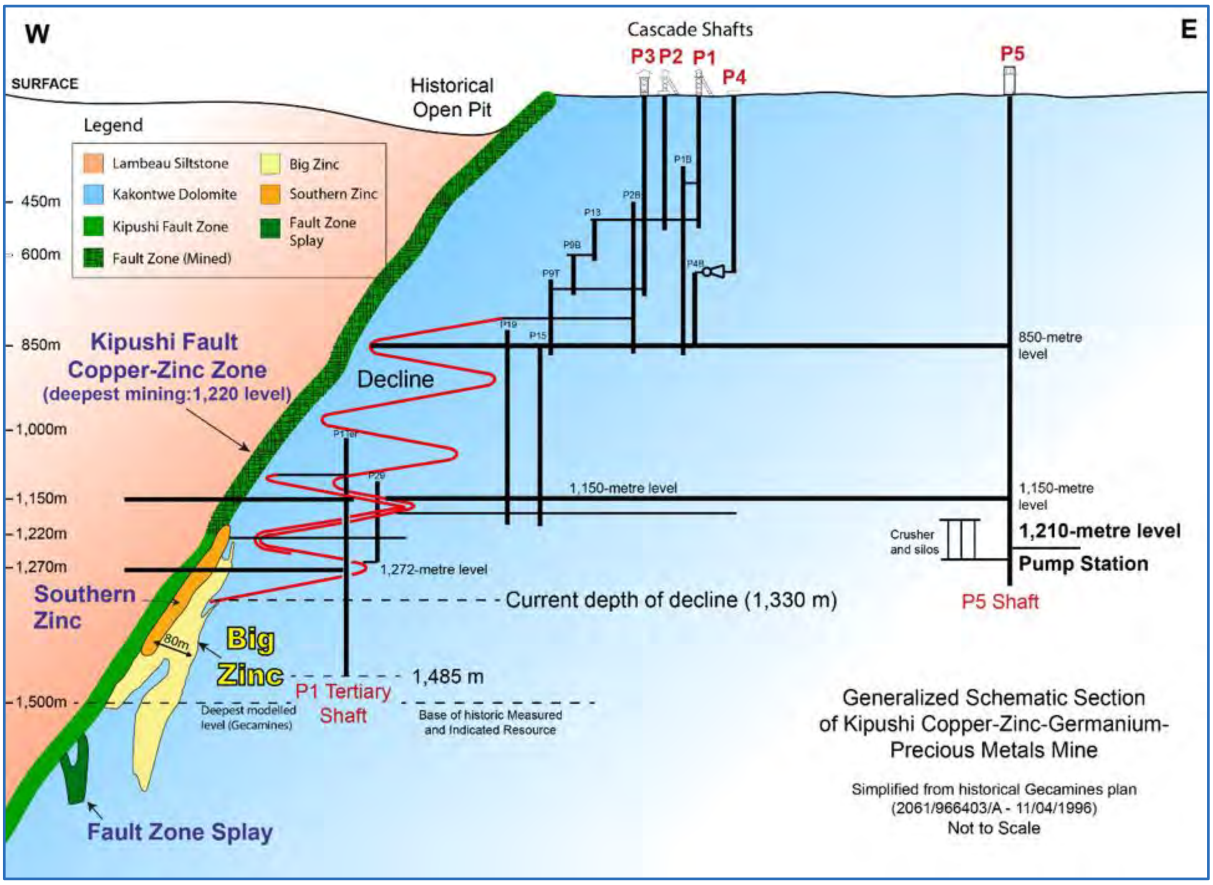
- Personnel access
- Material transfer
- Hoisting of excavated material

There is no decline access from the surface and all material and equipment will be transported to underground via the shaft. Major mobile equipment will be disassembled on surface into smaller components that can be handled within the shaft, then reassembled underground.

The 850 mL will be utilised as an intermediate level on the P5 Shaft to allow personnel to access infrastructure on that level, but the primary access for personnel and equipment will be on the 1,150 mL.

A schematic layout of the existing development is shown in Figure 16.41.

Figure 16.41 Schematic Section of Kipushi Mine



Ivanhoe, 2022

16.2.9.2 Future

Access to new work areas will be via a decline which will be extended from the existing face position at approximately 1,330 mL. The new decline is seen in light blue in Figure 16.35, with level accesses shown in grey to each of the mining levels.

On each level, the access leads to a perimeter drive (Figure 16.36, Figure 16.37 and, Figure 16.38), from which the ore drives are accessed. Once stoping commences, there will be no personnel access permitted beyond an open stope brow.

The second means of egress from the new mining areas will be through purpose excavated rises. These will tie into the existing second egress to surface via the Cascade Shafts, specifically via P15, P2 Bis and P2 Shafts (Figure 16.41).

16.2.10 Support of Excavation Voids

16.2.10.1 Stopes

Stopes are designed to the geotechnical parameters described in Section 16.1.7.

Stopes within the Big Zinc and Southern Zinc areas are required to be backfilled to maintain the structural integrity of the mine and allow for maximum recovery of the ore. Backfill has been specified as either Cemented Rock Fill or Rock fill depending on the strength required in that stope.

As Kipushi does not have sufficient waste rock to produce CRF, DMS discards are used to supplement the waste rock. In an attempt to reduce the surface stockpiles of waste rock and DMS, various blends of waste rock and DMS have been formulated to produce the CRF. By doing this, there will be sufficient waste rock until the last three years of the LOM.

Where appropriate unconsolidated fill will be placed, specifically where it won't be re-exposed by subsequent mining, including:

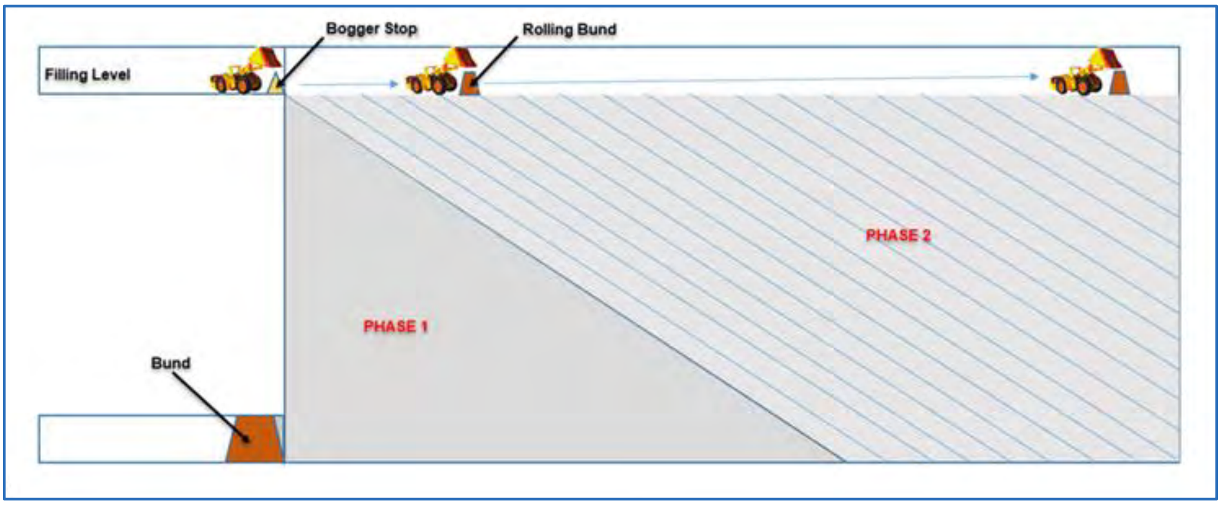
- Longitudinal stoping areas in the Southern Zinc zone.
- Upper secondary transverse stopes if the end walls of these stopes will not be exposed when mining subsequent stopes along the same production crosscut.

The path for this CRF is as follows:

1. Aggregates and cement slurry tipped down separate fill delivery lines from surface to underground fill plant.
2. Underground fill plant mixes CRF.
3. Mixed CRF is loaded into standard mine trucks via mixing plant.
4. Trucks haul from CRF plant to the upper level of the stope to be filled via the main decline.
5. Trucks tip CRF into CRF stockpile located on level perimeter drive.
6. Loader trams CRF from stockpile to stope.

Rock fill follows the same process except cement will not be added at the fill plant. The cement and rock fill process will happen in two phases. Phase 1 will be the CRF deposition in the stope from the top ore drive. The second phase will happen with the loader working over the filled material. In this phase, the loader stop will be removed and replaced with a rolling bund composed of waste material. The cycle will then repeat until the stope is filled (Figure 16.42).

Figure 16.42 CRF Placement Strategy



Mining Plus, 2019

For further detail on Backfill refer to Section 16.1.6.

16.2.10.2 Development

Development support design and costs are to the specifications described in Section 16.1.7. In addition to these specifications, rock bolt support is to be installed to the development face prior to the face being drilled. Mesh is also to be installed for S1, S2, and S3 ground conditions. The quality control for the resin bolt installation is set a maximum 4 mm annulus between drillhole and bolt rebar or effective mixing demonstrated through approved testing. However, a substitution for the resin bolt is allowed in the specification. Resin bolts may be replaced with full column post grouted DCP bolts for S0, S1, and S1a. The Byrnecut Offshore (BOPL) cost provision were based on this assumption.

16.2.11 Ventilation and Cooling

BBE Consulting South Africa (BBE) was retained by KICO to undertake a leakage audit at Kipushi, in order to establish the current status of the primary ventilation circuit and to make recommendations for improvement. BBE were provided with information on the mine design by KICO from which a 3D VUMA ventilation model was generated to simulate the ventilation and refrigeration requirements.

The initial basis of design was that a primary ventilation circuit suitable for the feasibility design and schedule would be able to be finalised once the design and schedule was completed. As such, flow volume required, and mobile equipment fleet numbers were not considered to be limiting factors when undertaking the design and schedule.

BBE created an initial 3D VUMA ventilation model in April 2018 based on the FS design at the time, which verified that the mine design and primary ventilation circuit proposed was suitable for the proposed mining method and extraction rates.

For further details refer to Section 16.7.

The operation will be relatively well mechanised with diesel equipment. The mine will use low sulfur fuel, CF1 low sulfur diesel 50 ppm, as defined by SANS 342:204. The diesel engines will be minimum Euro Tier II specifications. The diesel dilution criteria is 0.06 m^3/s per kW rated power at the point of use. The diesel fleet for the LOM is shown in Table 16.41.

Table 16.41 Diesel Fleet

Diesel Fleet	Power (kW)	Usage (%)	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
17 t LHD	275	60	2	4	4	4	4	4	4	3	3	3	3	3	3	3	3
51 t Dump truck	515	75	4	4	7	7	7	7	7	7	7	7	7	7	6	5	4
Dev drill	119	15	2	2	2	2	2	2	2	2	1	1	2	2	1	1	1
Production drill	110	15	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
UG grader	108	50	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Charge Up	110	40	2	2	2	1	1	1	1	1	1	1	1	1	1	1	1
4WD LDV	96	25	20	20	20	20	20	20	20	20	20	20	20	20	15	15	15
Cassette Utility	110	25	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Primary IT	156	50	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Secondary IT	115	50	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2

Mining Plus, 2019

The design case year will be 2025, which will have:

- Total rated diesel fleet: 7.95 MW
- Total rated corrected by utilisation factor: 4.32 MW
- Total LHDs and production trucks: 4.71 MW

The build-up with the time of the diesel equipment (total rated diesel fleet) will be:

- 2020: 5.97 MW
- 2021: 6.52 MW
- 2025: 7.95 MW
- 2032: 6.56 MW

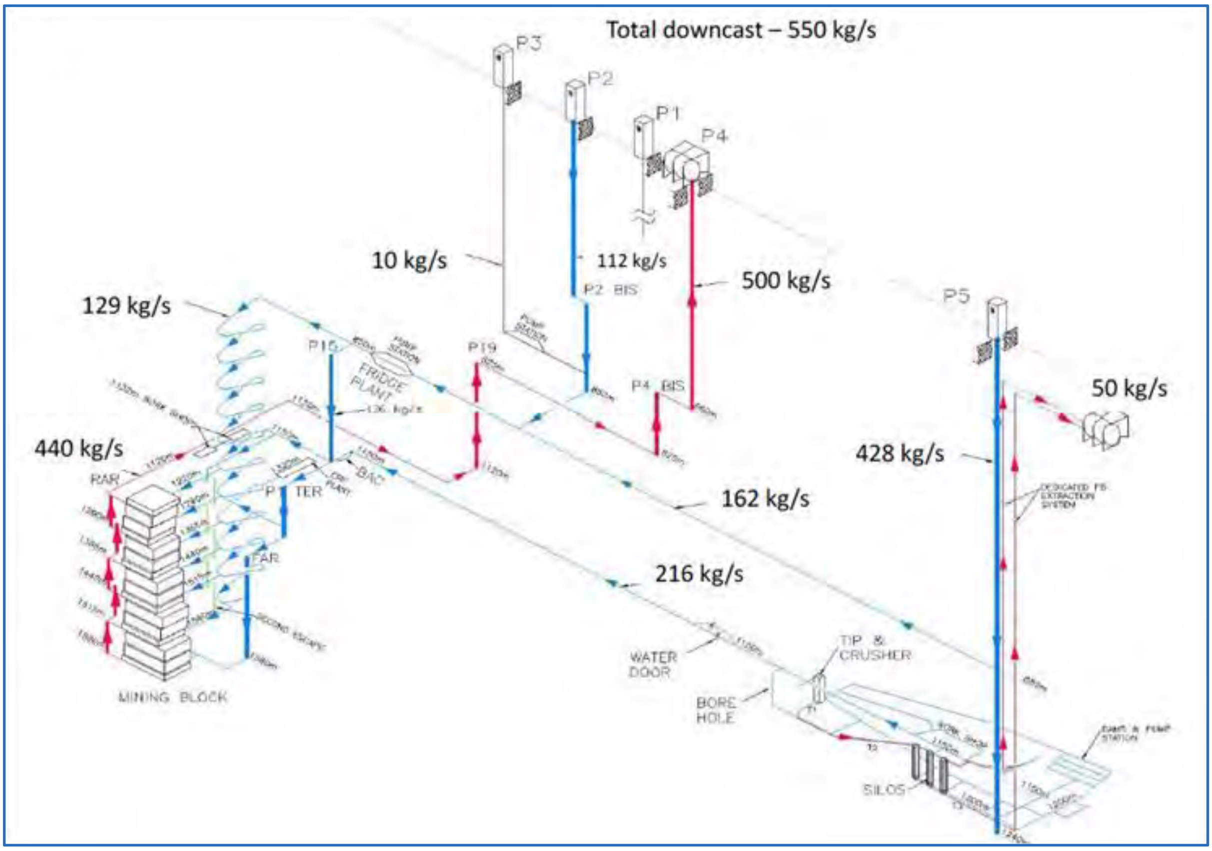
The heat load from diesel vehicles depends on how hard they are working e.g. hauling up decline or along level. For example, the total heat load applied in the modelling for diesel activity for year 2025 is 5.7 MW.

The largest power rated vehicle is the dump truck at 511 kW and at 0.06 m³/s per kW, these vehicles require minimum ventilation of 31 m³/s at point of use. The LHDs will require minimum ventilation of 16 m³/s at point of use.

The dump truck's last point of entry into a development area will be at the last through ventilation position i.e. there will never be a dump truck and an LHD in the same single heading.

To meet these criteria a primary flow of 550 kg/s is required. Figure 16.43 shows how this primary flow is distributed throughout the network.

Figure 16.43 Kipushi FS Primary Airflow

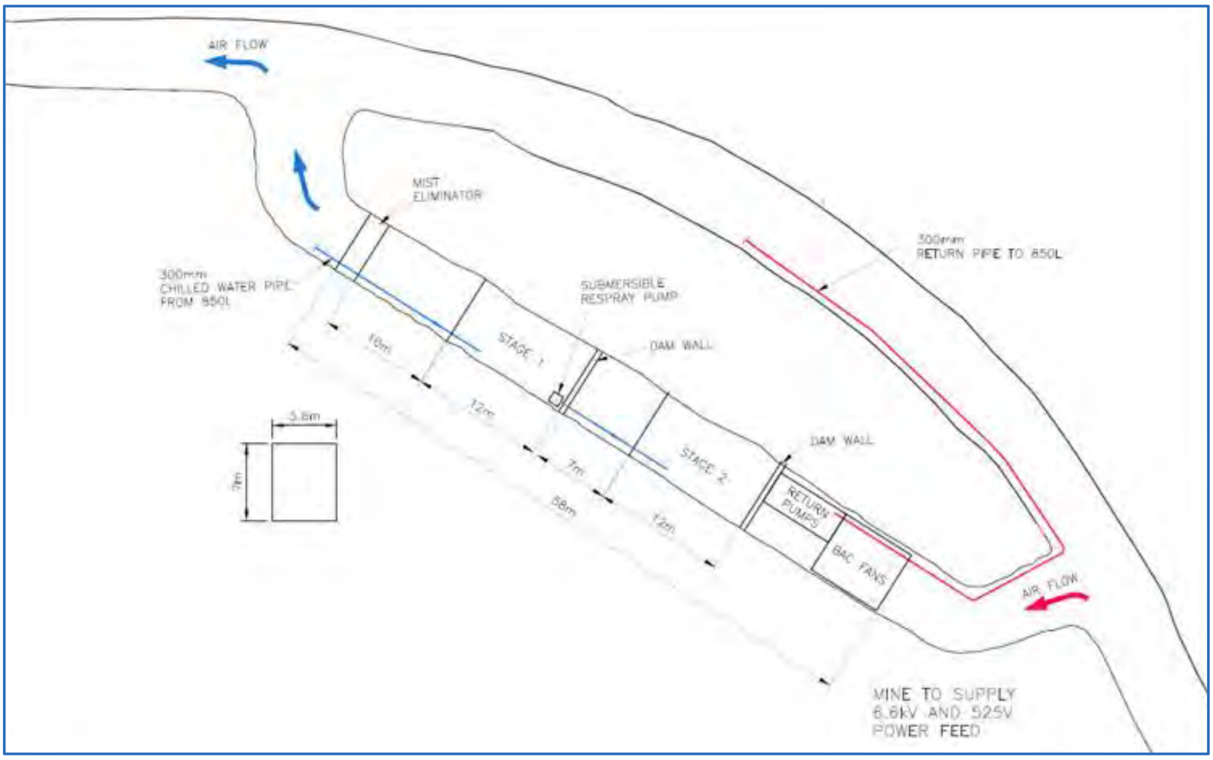


BBE, 2018

16.2.11.1 Primary Cooling

Different options were assessed for the location of cooling system components. The preferred option is underground Bulk Air Cooler (BAC) and underground refrigeration circuit. The BAC spray chamber will be located on 1,150 mL. Refrigeration machines will be in the in-plant chamber on 850 mL, with heat rejection into dewatering system and main pump station on 850 mL. Figure 16.44 shows the BAC conceptual layout on the 1,150 mL.

Figure 16.44 BAC Loop Conceptual Layout Showing Airflow Through Components



Mining Plus, 2019

16.2.11.2 Secondary Ventilation

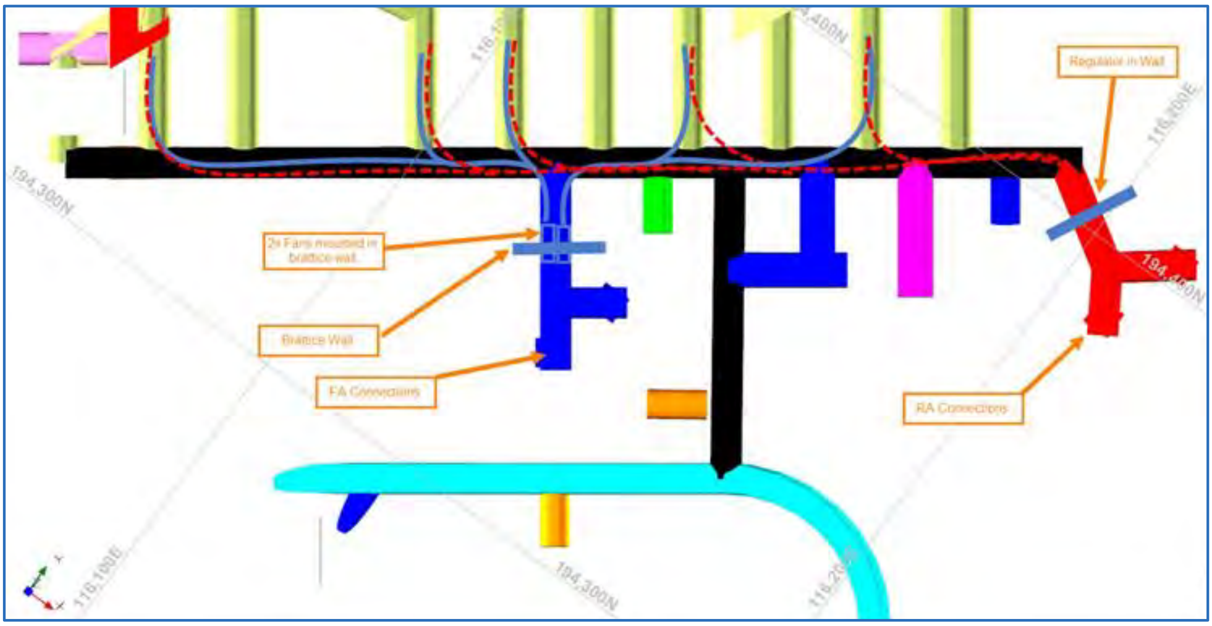
Fresh air will be directed to the working areas with flexible ducting. A typical level layout showing fan locations, and regulator positions are shown in Figure 16.45. Fans mounted in a vent wall will force air into duct running along the footwall drive, into the crosscuts, and up to the working areas. Used secondary ventilation air will exhaust back from the working areas to the return airway at the north-eastern end of the footwall drive.

16.2.12 Drill and Blast

Stope material will be broken using conventional longhole drill and blast. While it is expected that drill and blast methods will be refined during the course of the operation, for the purpose of the study the following is assumed.

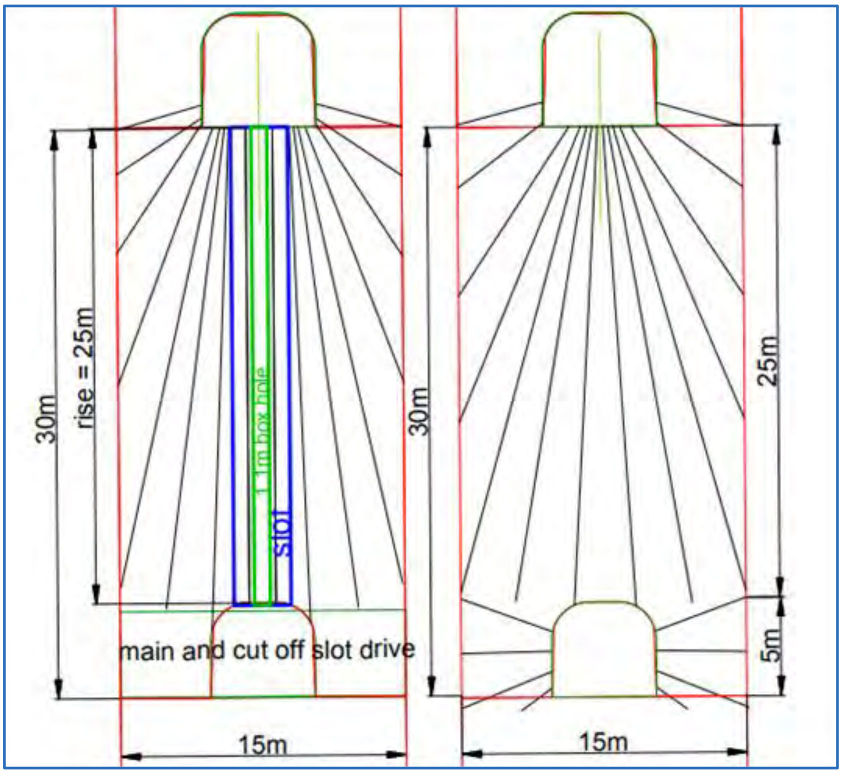
The majority of the holes in the primary and secondary stopes will be down-holes, drilled from the cross-cut above the stope with some horizontal holes drilled from the lower level (Figure 16.46).

Figure 16.45 Secondary Ventilation Layout – Typical Level



Mining Plus, 2019

Figure 16.46 Primary and Secondary Transverse Stopes – Slot and Ring Design



Mining Plus, 2019

The tertiary (sill) stopes will be blind up-holes drilled from the ore drive.

16.2.13 Material Handling

All material (ore and waste) will be moved by a trackless, diesel mining fleet, comprising of Sandvik LH517 loaders and Sandvik TH551 trucks.

The same trucking fleet will also transport CRF material from the underground CRF plant to the working levels, where it will be dumped into CRF rehandle stockpiles on the perimeter drives. From there, the LH517 loaders will place the fill material into the stopes.

The typical path for broken stope material will be:

- LHD moves material from stope draw point to either a footwall drive stockpile or truck loading bay.
- LHD loads truck in truck loading bay between access and footwall drive.
- Truck hauls up decline to crusher on the 1,150 mL.
- Material is crushed and conveyed to the crushed ore storage bin.
- Material is loaded into skip and hoisted to surface.

At the end of the stope cycle the brow of the stope will be open and the remaining material will be recovered by remote loading.

Development material will follow the same path as stope material but will be loaded to the nearest stockpile (not necessarily on the footwall drive), typically within 150 m of the development face.

A significant portion of the development material is not economic to process so is classed as waste. This waste is eventually placed in stope voids as CRF but must first be hoisted to the surface for crushing and screening.

16.2.14 Dewatering

The de-watering plan comprises two parts, the secondary system which de-waters the areas under development and the primary system which pumps these and inflows from existing excavations to surface.

For further detail on dewatering, refer to Section 16.8.4.

16.2.14.1 Primary Dewatering System

The existing dewatering system forms the backbone of the future system. The inflows and the corresponding infrastructure to pump these flows are as follows:

- Inflows above 850 mL:
  - 970 m^3/h initially, dropping to 770 m^3/h over the LOM (270 to 210 L/s).
  - 850 mL Pumping Station lifts water to surface dams via P5 Shaft.
- Inflows between 850 mL and 1,150 mL:
  - 790 m^3/h initially, dropping to 540 m^3/h over the LOM (220 to 150 L/s).
  - 1,200 mL Pumping Station lifts water to surface dams.
- Inflows below 1,200 mL (Big Zinc and Southern Zinc mining areas):
  - 540 m^3/h initially, increasing to 1,220 m^3/h once the lowest mining levels are accessed, and then reducing to 1,100 m^3/h by end LOM (150 to 340 to 300 L/s).
  - Water lifted up via Peerless pumps in P1 Ter Shaft to Gallery 19 Dam and flows by gravity to 1,200 mL Pumping Station.
  - 1,200 mL Pumping station lifts water to the surface dam.
- Total delivery from 1,200 mL Pump station:
  - 1,330 m^3/h initially, increasing to 1,770 m^3/h once the lowest mining levels are accessed, and then reducing to 1,600 m^3/h by end LOM (370 to 490 to 450 L/s).

16.2.14.2 Secondary Dewatering System

The secondary dewatering system for Big Zinc and Southern Zinc mining areas will deliver water to the existing primary dewatering circuit for pumping out of the mine.

The secondary system delivers inflow water to the 1,327 mL for pumping via the Peerless pumps installed in P1 Ter shaft.

- As the decline is advanced sumps will be excavated at regular intervals, approximately every 30–40 vertical metres. Drain-holes will connect each decline sump to the level below.
- Sumps will also be included on each mining level, at a low point of the perimeter drive. Drain-holes will connect each level sump to the level below.
- A range of submersible pumps will be utilised as required, with predicted pump motor sizes of:
  - 8 kW
  - 20 kW
  - 37 kW
  - 90 kW

These pumps will pump water from sumps and any downgrading faces up to a staging pump system.

Permanent staging pump stations will be established on the Big Zinc decline every 60–80 vertical metres, to stage pump inflow water up to the 1,327 mL. These pumps will be skid mounted for portability and incorporate a water hopper. It is anticipated that these will be centrifugal 220 kW pumps.

16.3 Underground Infrastructure

16.3.1 Existing Underground Mine Infrastructure

The existing mining infrastructure consists of five surface vertical shafts and a number of sub-vertical shafts allowing access to deeper levels. The shafts included in the Kipushi 2022 FS planning are:

- Shaft 1 (0–650 mL): Second egress
- Shaft P1 Bis (400–850 mL): Second egress
- Shaft P1 TER (1,138–1,480 mL): Second egress
- Shaft 2 (0–500 mL): Ventilation exhaust
- Shaft P2 Bis (500–850 mL): Return ventilation
- Shaft 3 (0–740 mL): Second egress
- Shaft 4 (0–650 mL): Ventilation exhaust
- Shaft 4 Bis (650–825 mL): Return ventilation
- Shaft 5 (0–1,240 mL): Personnel, material, services, rock hoisting, and ventilation
- Shaft P9 (700–1,010 mL): Second egress
- Shaft 15 (850–1,172 mL): Second egress
- Shaft 19 (825–1,120 mL): Return ventilation

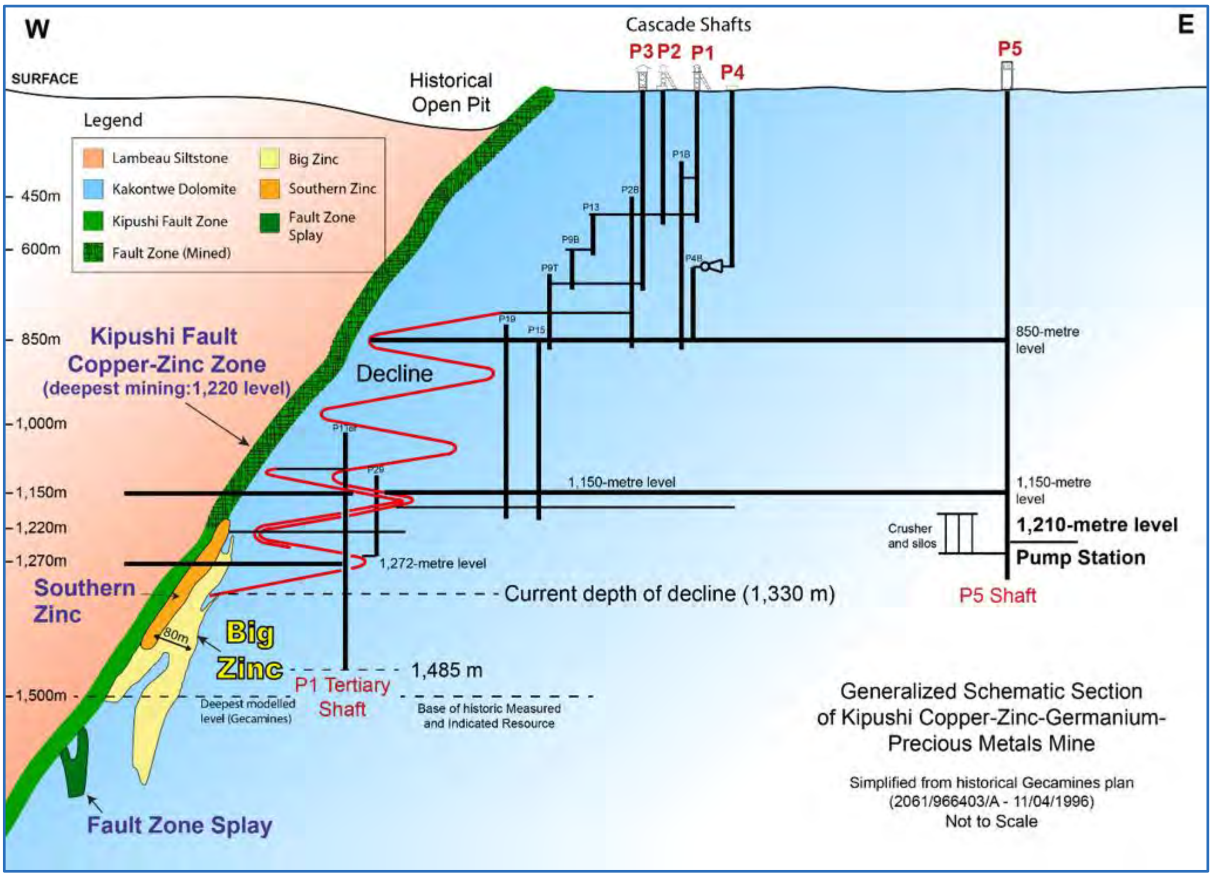
The 850 mL will be utilised as intermediate level on the Shaft 5 to allow personnel and equipment to enter the mine workings without doing so via the main haulage and crusher level, minimising interactions, and downtime to the haulage network.

The main working area is connected to Shaft 5 via the 1,150 mL main haulage level. There is a crusher chamber at 1,200 mL; the crusher level is now dewatered. The underground infrastructure exposed since dewatering, is in relatively good order. The crusher has been replaced and commissioned.

A 5 m H x 5.8 m W decline was developed from 725 mL to approximately 1,330 mL, the upper to deeper working levels and the top of the Big Zinc.

A schematic layout of the existing development is shown in Figure 16.47.

Figure 16.47 Schematic Section of Kipushi Mine



Ivanhoe, 2022

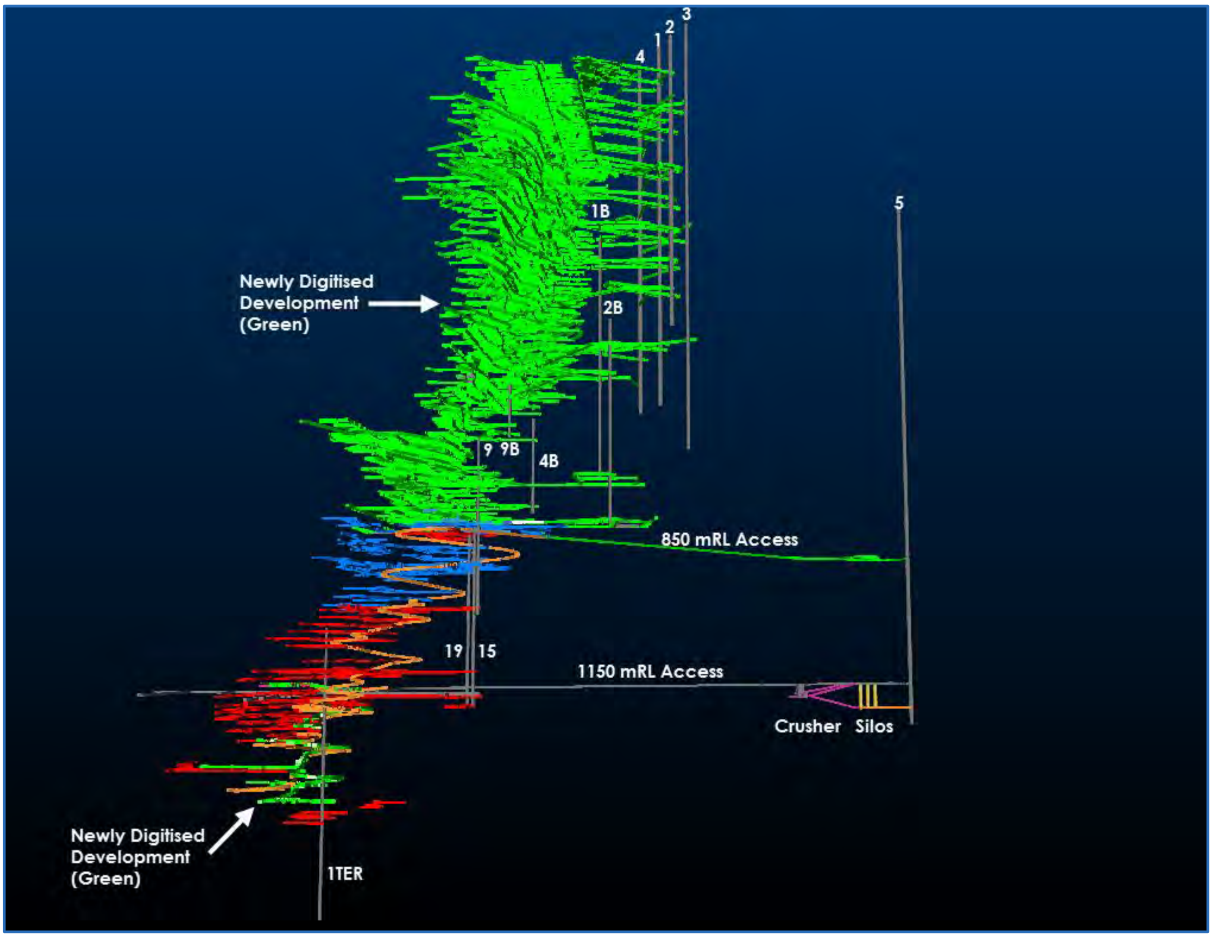
Digitised shafts, decline and existing development from the 855 mL to the 1,347 mL is shown in Figure 16.48.

A network of underground pumps, cascading dams and pipework currently dewaters the mine at a maximum rate of 3,500 m^3/h. Water is pumped from shafts and sumps to intermediate settling dams on the 1,200 mL, 1,150 mL, 1,112 mL, and 850 mL levels and then to surface via Shaft 5.

All five 3.5 MW-12 stage pumps at the 1,200 mL pump station have been replaced and commissioned. A new pump station with three 3.2 MW-8 stage pumps at the 850 mL has been constructed and commissioned to replace the old cascade pumping system. The latter being kept for emergency use.

Workshops and magazines exist on the 1,132 mL and 850 mL levels. These areas require rehabilitation but will provide locations for machine maintenance, breakdown areas, welding bays, wash bays, tyre changing and storage, explosives storage, lubricant tanks, and diesel storage. A workshop at 1,150 mL is now fully operational and equipped with new 5 t and 20 t cranes.

Figure 16.48 Digitised Existing Development



OreWin, 2017

16.3.2 Underground Rock Handling

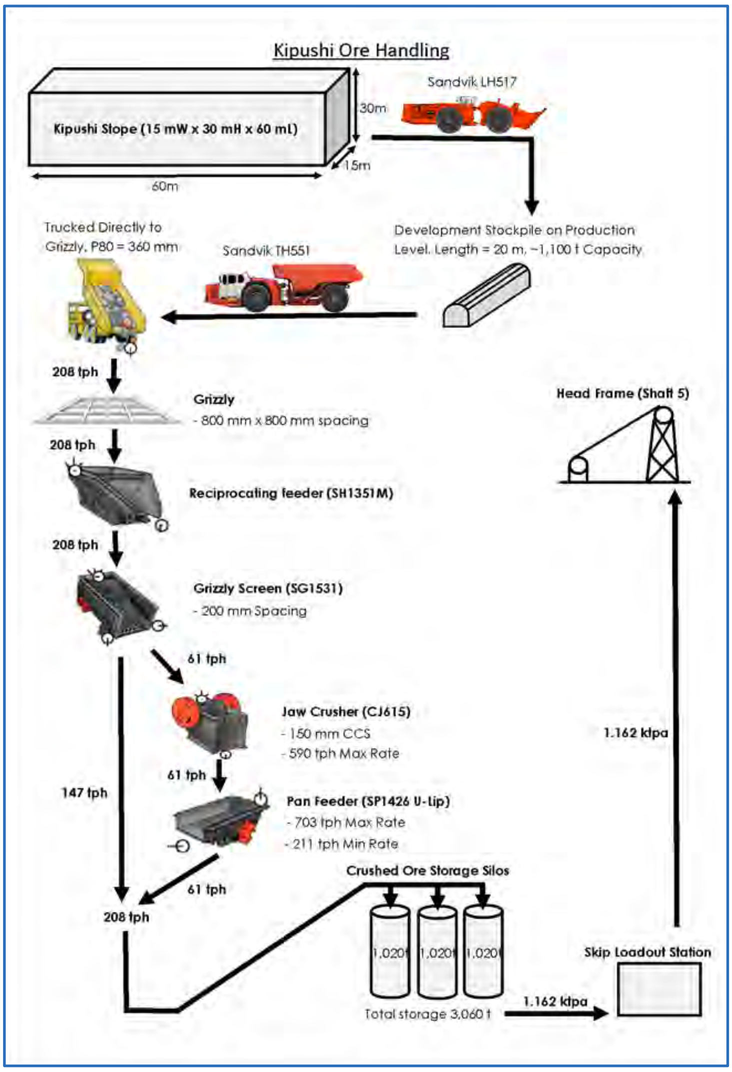
All material in the Big Zinc and Southern Zinc mining areas will be moved to the surface via the P5 shaft. Waste will be returned to the underground for backfilling through raises to the backfill plant. Potential underground storage of waste is also considered in the design.

On every level of the mine, ore and waste material are loaded with loaders (LHDs) to stockpiles. The material is then loaded onto trucks which transport material up the decline and dump into an 800 mm x 800 mm grizzly located at the 1,150 mL. From there, material flows to a reciprocating feeder and eventually to a 200 mm screen. Passing material (<200 mm) is sent straight to ore bins. The remaining material is sent to an underground jaw crusher. The material crushed at this stage is delivered to a plate feeder for storage in the ore bins. The material is then conveyed to Shaft P5 where it will be hoisted to the surface.

On the surface, the material can be diverted through conveyors directly to the process plant or to a waste rock stockpile prior to reclamation for processing.

The recommended underground materials handling design process flow is shown in Figure 16.49.

Figure 16.49 Underground Materials Handling Process Flow



Mining Plus, 2019

16.3.3 Haulage to 1,150 mL

An existing decline extends to approximately the 1,330 mL, and the FS design incorporates an extension of that decline to the lowest level of extraction at 1,590 mL. All material is planned to be trucked from the working levels up to the 1,150 mL.

Refurbishment works on the 1,150 mL have largely been completed. Rails have been removed and the floor concreted for conversion to truck haulage, with lined drains incorporated to the side.

16.3.4 Crushing Facilities

The existing crusher chamber and accompanying excavations of 1,150 mL have been rehabilitated. Kipushi Corporation has installed a Sandvik CJ615 Jaw Crusher.

Based on similar operations that utilise underground jaw crushers and similar rock properties, an 800 mm x 800 mm grizzly spacing is assumed. Kipushi Corporation requested that the underground crushing stage to produce at least 80% of material smaller than 250 mm (P_{80}=250 mm). Simulations were done on the Sandvik CJ615 Jaw Crusher with material similar to the Kipushi Ore (900 mm feed size, work index of 16, a density of 4.00 t/m³ and moisture content of 5%) show this is within the test parameters.

A trade-off study for an underground crushing station was undertaken which resulted in the decision to not include an underground crushing chamber.

16.3.5 Material Properties

The in-situ ore density ranges between 3.20 t/m³ and 2.80 t/m³ with an average density of 3.60 t/m³. Kipushi waste rock has an average density of 2.94 t/m³. The various rock types densities are provided in Table 16.42.

Table 16.42 In-Situ Material Densities

Rock Type	Density Results (t/m³)		
	Minimum	Mean	Maximum
Dolomite	2.82	2.85	2.89
Siltstone	2.70	2.75	2.79
Sandstone	2.70	2.75	2.83
Shale	2.76	2.82	2.86
Sphalerite	4.11	4.20	4.38
Sulfides	3.87	4.03	4.22

16.3.6 Capacity

The design capacity for the Kipushi 2022 FS is 0.81 Mtpa ore and 0.36 Mtpa waste for a total production of 1.12 Mtpa. A moisture content of 5% has also been assumed.

The nominal minimum throughput of the underground materials handling system to reliably feed the plant is determined as follows:

$$R=([C\times10]\ \hat{}6\times(1+m))/(\mu\times N)$$

Where:

R=Nominal Design Rate in t/h.

C=Design capacity in Mtpa (dry) of the processing plant.

M=Moisture content (% weight for weight) = 3% for design purposes.

µ=utilisation of the system as a percentage.

N=Total number of hours per year = 8,760.

The underground materials handling system has an expected system utilisation of 72.5% based on the transference of ore from the underground crushing station tipple through to the surface conveyors. This equates to a nominal rate of 200 t/h.

The required design rates of the materials handling streams should be in excess of the nominal rates to account for spring capacity. The design rates of the materials handling streams should be of 300 t/h.

The maintenance strategy, operational strategy, and equipment selection shall ensure the stated availability and reliability parameters are achieved as a minimum.

16.3.7 Waste

The FS has investigated options for underground storage of waste to reduce the operating cost associated with hoisting the waste to the surface before returning it to underground. One of the alternatives under consideration is to make use of an underground crusher that will process the underground development waste rock and store it in excavated voids to be used when required for the production of CRF to be used for stope filling.

The use of existing excavations will be investigated once additional survey pickups have been completed.

16.3.8 Workshops

Multiple workshops will be utilised in the mining of Big and Southern Zinc. A mixture of fixed and mobile workshops will be distributed throughout the mine to enable efficient maintenance in the mine.

List of underground workshops as follows:

- Main Underground Workshop.

  The main underground workshop is at 1,150 mL is now fully operational and equipped with new 5 t and 20 t cranes. All light vehicle servicing and all major equipment rebuilds will occur in the main underground workshop.

  An underground crib room is to be in the main workshop area.

- Drill Workshop.

  A secondary drill workshop will be located adjacent to and opposite the main workshop at 1,134 mL.

- Reassembly Workshop.

  The reassembly workshop is located near the base of Shaft P5 for the purpose of assembling equipment as it is transported into the 1,150 mL.

- Fuel and Lubricant Storage.

  The fuel bay is located on the 1,112 mL. Distribution of diesel fuel to underground will be via a diesel range down Shaft P5, along 850 mL haulage downgraded towards Shaft P15, and down Shaft P15 to the 1,112 mL Diesel Bay. The fuelling station will have the storage tanks and pumps installed in an enclosed drift with fire doors and appropriate fire suppression systems.

Currently, there is some trackless equipment that was slung down the shaft on early stages of the mine. The current plan is for the workshop to be able to carry out the reassembling of this equipment along with all the regular workshop operations. Further reassembly work will be carried out in the Reassembly Workshop.

16.3.8.1 Drill Workshop

A secondary drill workshop will be located adjacent to and opposite the main workshop at 1,134 mL.

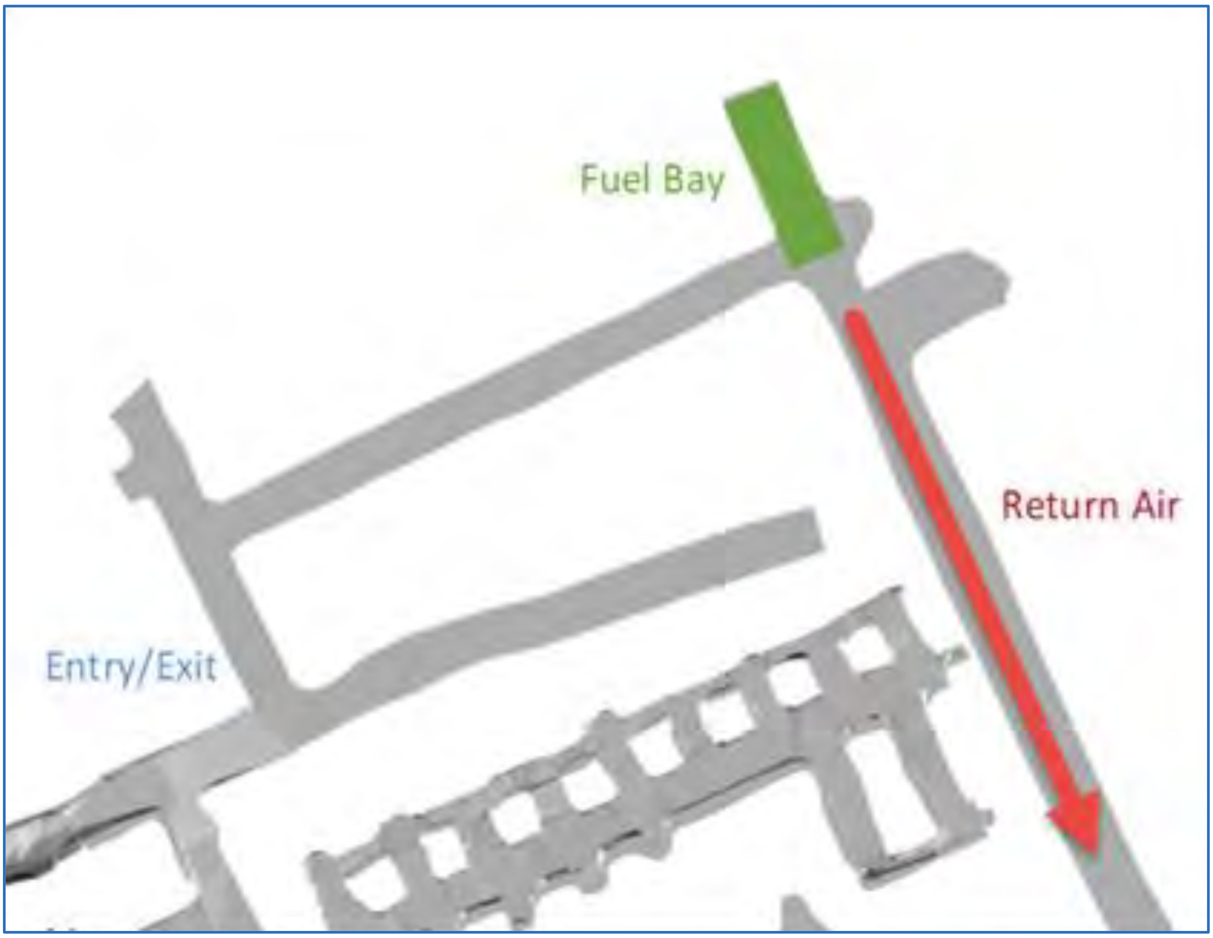
16.3.8.2 Reassembly Workshop

The reassembly workshop is located near the base of Shaft P5 for the purpose of assembling equipment as it is transported into the 1,150 mL.

16.3.8.3 Fuel and Lubricant Storage

The fuel bay is located on the 1,112 mL as shown in Figure 16.50.

Figure 16.50 1,112 mL Refuelling Bay



Mining Plus, 2019

Distribution of diesel fuel to underground will be via a diesel pipeline down Shaft P5, along 850 mL haulage downgraded towards Shaft P15, and down Shaft P15 to the 1,112 mL Diesel Bay. The fuelling station will have the storage tanks and pumps installed in an enclosed drift with fire doors and appropriate fire suppression systems.

The refuel station will be fitted out by installing a bunded concrete slab, sized for both the storage tanks and installed pumps and the vehicles to be refuelled. Any spillage is to be captured on the concrete slab and directed to a sump where oily water will pass through an oily water separator before being pumped to the decline sump.

The enclosed drift should also be equipped with fire doors, foam type fire extinguishers in order to provide an appropriate fire suppression system.

Raw water is to be provided at the refuelling area for hose down and clean up purposes.

16.3.8.4 Crib Facilities

An underground crib room is to be in the main workshop area.

16.3.9 Explosives Storage

The main explosives magazines will be underground, rather than on the surface, with a fenced, lockable compound will be included on surface at the P5 Shaft area. Explosives will be delivered as follows:

- Explosives delivery vehicles will drive into the compound, which will then be locked.
- Explosives will be unloaded into an explosives transfer shed for temporary storage, before being transported to P5 Shaft for delivery to the underground workings as soon as is practicable.
- Once transported underground to 1,150 mL, explosives will be subsequently delivered to the 1,170 mL and 1,182 mL magazines.
- Dedicated explosives carrying light vehicles will transfer explosives from underground magazines to the work areas as required.

The Main Magazine will be located on the 1,182 mL and will house all high explosive products, detonator, primers, and packaged explosives.

An emulsion magazine located on the 1,170 mL will store the emulsion chemicals. The emulsion magazine is located close to the Main Magazine for convenience.

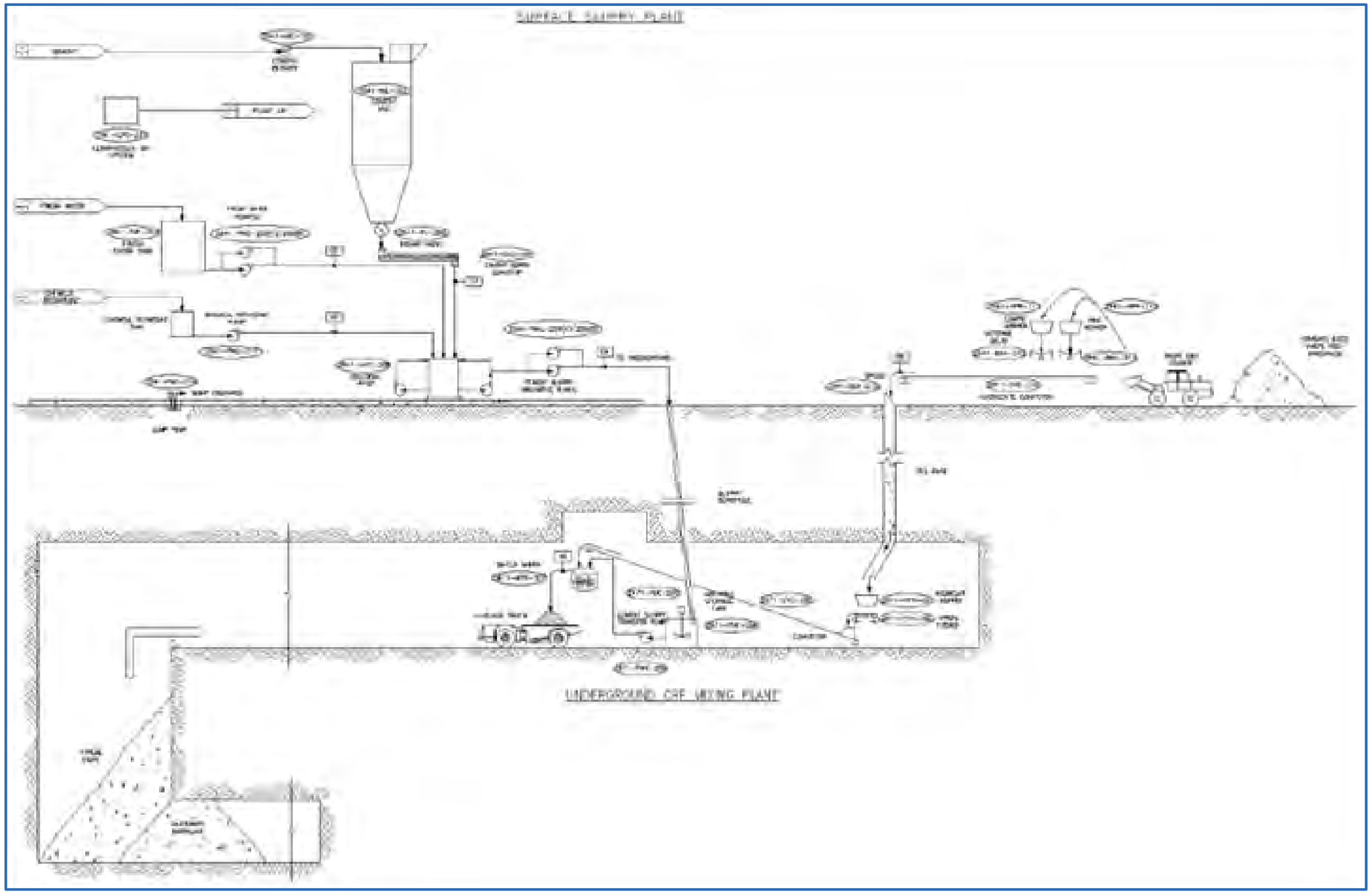
16.3.10 Backfill Facilities

The primary stopes require backfilling with CRF and the secondary stopes with rock fill (RF).

The CRF process flow is shown in Figure 16.51.

The CRF facilities will consist of a surface crushing and screening plant, surface cement plant, underground CRF mixing plant, and the option of an underground crushing and screening plant. On surface the cement plant, crusher, and screening will be located in close proximity to each other.

Figure 16.51 CRF Process Flow Sheet



Mining Plus, 2019

16.3.10.1 Cement Storage and Delivery

Regular Portland cement will be stored on the surface in silos (sized for one-month capacity) that will be fed by cement trucks coming from local suppliers. The colloidal mixing plant on the surface combines clean water from a clean water tank, cement from the two cement silos, and a chemical hydration retardant from a chemical retardant tank into a slurry consistency and it then discharges this slurry to an underground agitated tank via boreholes. Depending on the requirements for the CRF strength, the cement slurry may be mixed with dewatered concentrator tailings to reduce the final size of the Tailings Storage Facility (TSF). As not enough waste will be generated during development, DMS reject material will be required to supplement the waste stream.

The plant utilises two 300 mm borehole lines, one main line and a second as backup. The cement slurry lines from the surface are located into backs of batch plant chamber and second in the drive to the rear of the CRF plant. The cement slurry lines are drilled from surface by directional drilling with accuracy better than 1%. These 300 mm holes will be drilled, and locations confirmed before construction of the CRF plant. The slurry is stored underground in an agitated tank until being pumped into a twin-shaft continuous mixer containing crushed and screened waste rock.

16.3.10.2 Crushing and Screening Plant

A surface plant will be constructed at the start of the project with an underground plant constructed later. Waste material will be used by the crushing and screening plant to produce two sizes of material for use in the CRF plant or as road base in the underground.

As only a surface plant (and one hoist) is available ore and waste will be trucked to the P5 Shaft (in campaigns as only ore or only waste) and hoisted to the surface.

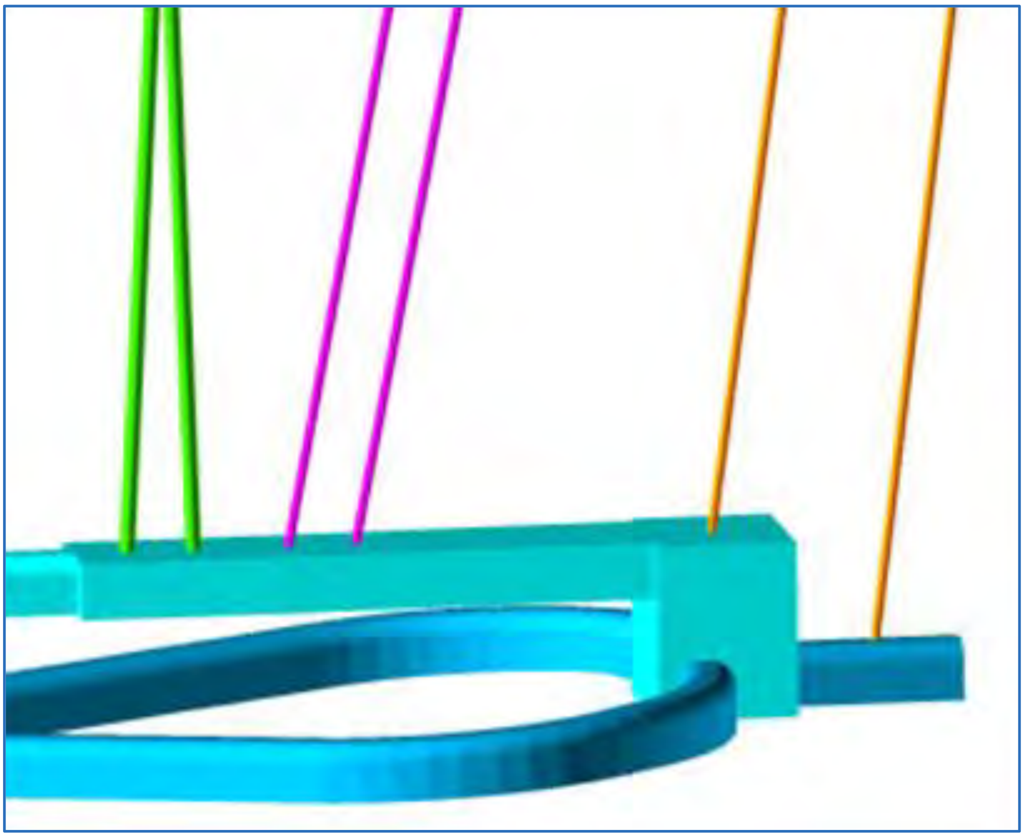
The waste balance for filling requires that additional material be sourced to fulfil the backfill requirements, especially in later years. The 1,150 mL waste transfer level can also contain a series of waste stopes to provide this material.

16.3.10.3 Surface Crushing and Screening Plant

A surface plant will be constructed at the beginning of the project and will be located close to the surface cement facility. The material will be crushed into 10–76 mm and sub 10 mm sizes. Two large (3–4 m) boreholes will transfer screened material to the CRF plant underground Figure 16.52. These boreholes are to be drilled in two passes.

The target location of these holes is critical, so the design uses a two-stage drilling design to improve accuracy. Surface to 850 mL and 850 mL to mixing plant conveyor at 1,260 mL. Aggregate can be accessed from the 850 mL if required for road base or stemming on the upper levels. Three boreholes will be required, one for surface aggregate, one for cement, and one for underground aggregate.

Figure 16.52 Boreholes into 1,320 mL CRF Plant



Mining Plus, 2019

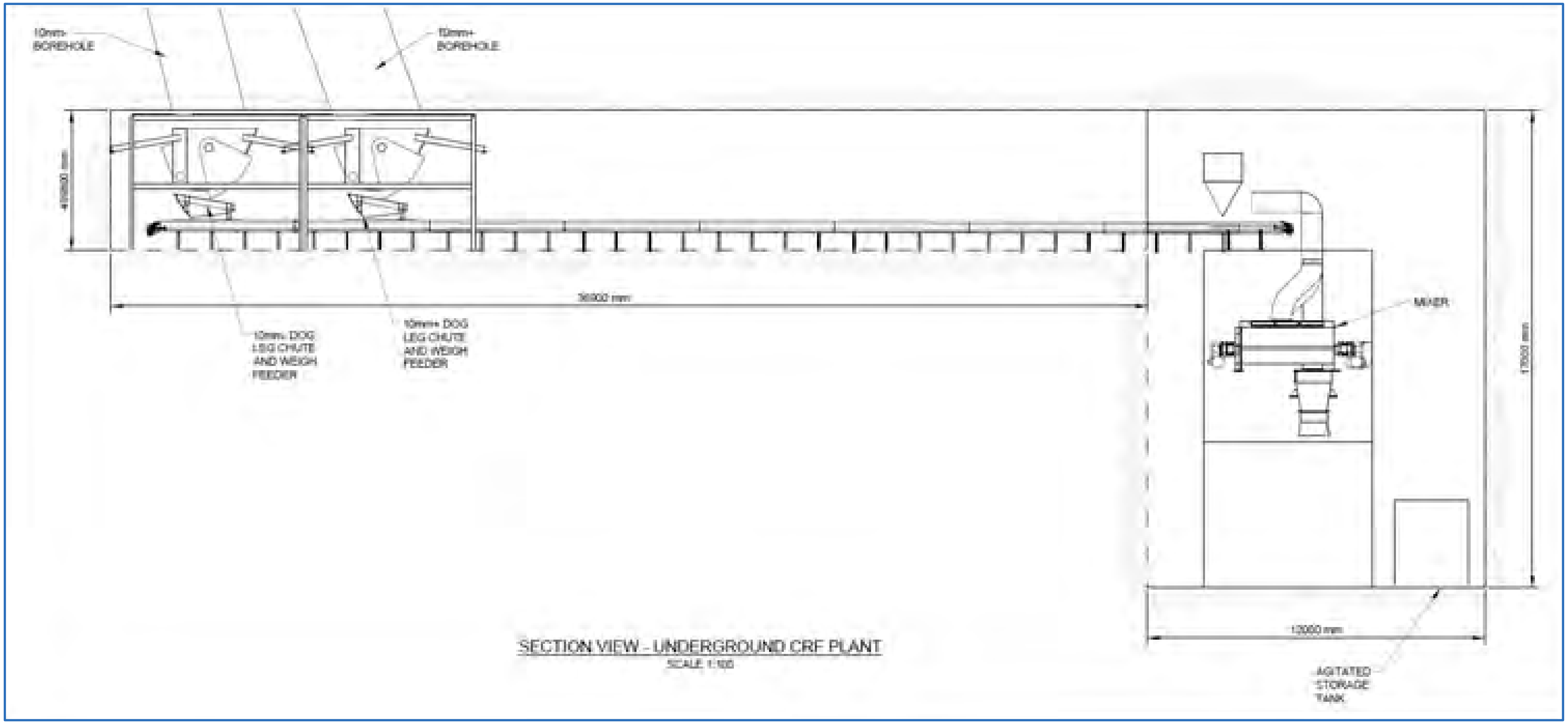
16.3.10.4 CRF Plant

The CRF plant will be located on the 1,320 mL. The plant is located to allow boreholes from the surface in a single pass. Twin cement slurry lines from surface will intersect the plant (one in the backs of batch plant chamber and the other in the drive to the rear of plant). Both of these holes are single-pass from surface requiring directional drilling. These lead to an agitated storage tank.

Figure 16.53 shows the underground layout for the aggregate and cement mixing and discharge into dump trucks. A weigh feeder reclaims the rock from each raise and transports it to the twin shaft mixer via a conveyor. The slurry flow rate from the underground agitated tank into the twin shaft mixer is metered using the weight of the waste rock on the conveyor. The truck simply drives underneath the mixer for loading. Trucks will transport CRF material to backfill stockpiles located on each level.

The development design allows for the supply of the waste material from both surface and underground.

Figure 16.53 Underground CRF Plant – Sectional View



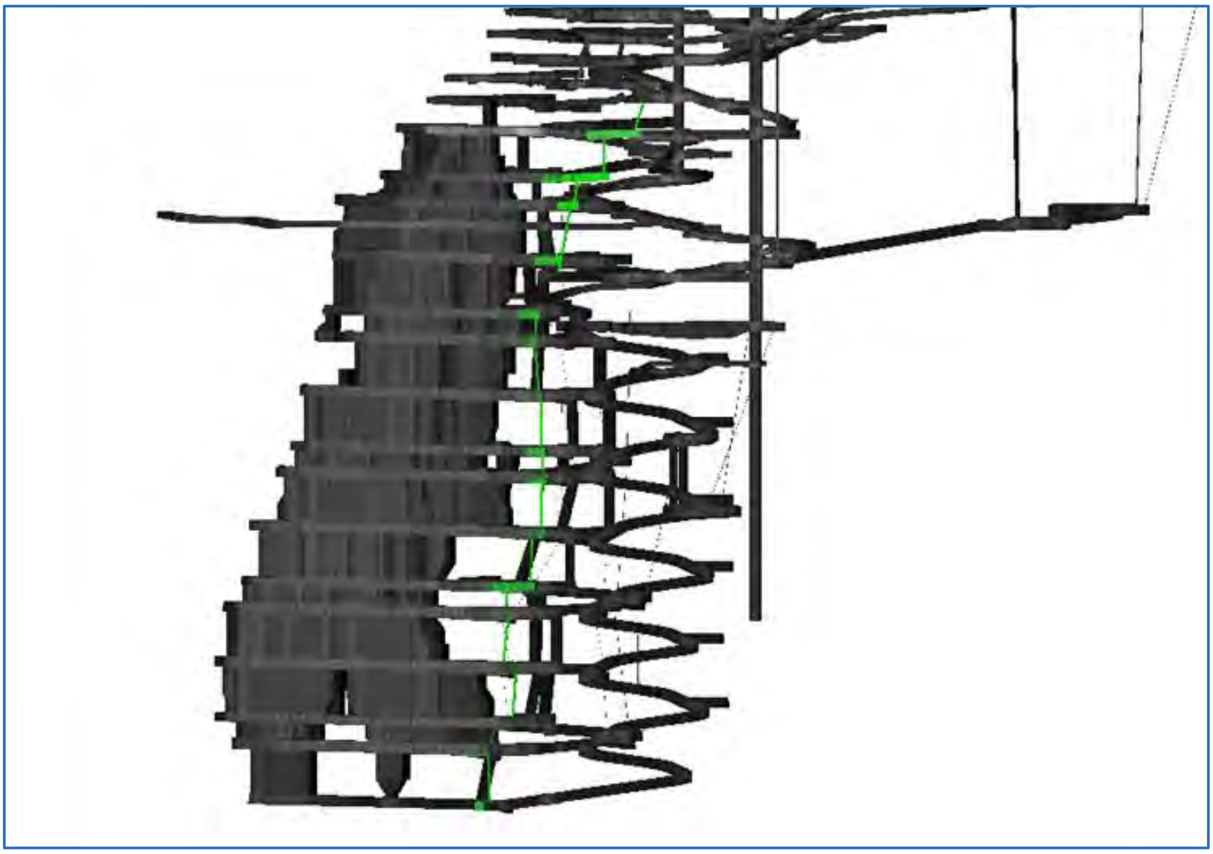
Mining Plus, 2019

When CRF is not required, such as when backfilling secondary stopes (rock fill only), the cement mixing system is inoperative and waste rock bypasses the twin shaft mixer. Road base can be accessed in the same way by running the plant with only one weigh feeder operating. This saves hoisting road base from the surface.

16.3.11 Secondary Egress

A dedicated escapeway has been included in the design, linking the perimeter drive on each level as shown in Figure 16.54. The top of the escapeway links to the current mines established escape system.

Figure 16.54 Escapeway System



Mining Plus, 2019

16.3.12 Refuge Chambers

Emergency refuge forms an integral part of a wider Emergency Response Plan. Fires, explosions, rock-falls, flooding, and the release of smoke and other forms of toxic gas resulting in an irrespirable atmosphere are typical incidents that occur where the need for employee safe refuge is required.

Refuge chambers will be located at various strategic sites throughout the mine where protection to persons is possible should such an emergency arise.

16.3.13 Mine Services

16.3.13.1 Water

Raw water is transported from Shaft 5 water tower, by gravity, and made available for mobile equipment (fire wagons, water bowsers) as well as to the raw water supply tank (500 m^3), that can also be used as a firewater tank. From the raw water tank, water is used for surface dust suppression, as well as filtered gland service water for slurry pumps. Metallurgical testwork conducted indicates that the raw water recovered from mine dewatering is not appropriate for use in the flotation circuit. Therefore, potable water is utilised as process water make-up. Provision is made, however, for make up using raw water if required.

Potable water is received from the local municipal supply and stored in the plant in a potable water tank (50 m^3) for further water distribution.

16.3.13.2 Compressed Air

As part of the design criteria, no compressed air will be piped to working areas, the mobile fleet will have onboard compressors, where required. Refuge chambers will be required to have an air compressor system installed. A duty and standby compressor system will be installed at the main 1,132 mL workshop.

16.3.13.3 Underground Electrical

The mine site is supplied from the network grid at Lubumbashi via two overhead power lines (110 kV and 50 kV). There are currently two emergency power generators with an operational capacity of 2 Megawatt. These generators supply backup power for emergency hoisting and primary ventilation. There is currently no backup system in place to supply power to the dewatering network in the event of a power outage. Given the criticality of underground dewatering requirements, a review of available gen-set power should be instigated as a priority.

Within the mine, power reticulation will be as follows:

- High usage electrical equipment (primary pumping, main fan station, refrigeration plant etc) are supplied by a 6.6 kV high voltage line.
- The existing underground low voltage reticulation comprises of 550 volts and 220 volts.

Electrical reticulation in future mining areas is planned to be at 1,000 volts, including the supplying of underground drill rigs, secondary ventilation fans, lighting panels, and face dewatering pumps (temporary systems).

The future mining areas will be supplied with power stepped down from the high voltage underground line using underground transformers, located on every third level. Cabling from these transformers will be run in vertical drillholes between levels wherever possible to reduce the 1,000 volts cabling distances.

16.3.14 Underground Communications

16.3.14.1 Communication

The mine communications will be managed through a VHF leaky feeder system. A fibre optic network is also to be installed in the new areas of the mine.

16.3.14.2 Firing Systems

The mine is to use firing line to designated firing locations for initiating development and stoping blasts.

16.3.14.3 Safety Systems

In underground mining environments, the risk of collision between machinery and personnel remains high where poor visibility and working in confined spaces can affect the ability of personnel to detect potential hazards before they occur. Through normal day to day activities between underground vehicles and mining personnel, there exists a high chance of personnel injury and high financial cost of damaged vehicles and productivity loss from any downtime occurred. In many cases, these collisions are a major cause of mine fatalities and the impact on the mines business operations can be phenomenal if steps to avoid collisions are not addressed.

In order to reduce this risk, proximity detection systems will be fitted to all underground equipment and personnel cap lamps. This system alerts heavy machinery and vehicles when personnel and other vehicles are in the vicinity to reduce the chances of interactions and incidents. The proximity system will use the fibre optic network and RFID technology to operate.

16.4 Mine Schedule

16.4.1 Schedule Strategy

Mining activities are scheduled in the following order of importance:

1. Mine sufficient ore to fill the process plant and maintain a two-month stockpile.
2. Always mine the highest grade (value) material that can be accessed at that time.
3. Only mine development when it is required to access production areas (subject to practical operation of the mining fleet), so as to defer expenditure on development where practicable.

Beyond these overarching objectives, there were additional scheduling constraints to be adhered to:

- Months 1–6: Early Works

  The mining schedule commences with a six-month period of Early Works, comprising rehabilitation of existing workings, and upgrading of facilities to enable the efficient operation of the Big Zinc and Southern Zinc mining areas.

- Months 7–12: Feasibility Study Schedule

  This period signifies the start of the full mining contract, post-Early Works. This includes the commencement of the extension of the decline to access the Big Zinc mining area, plus also mining of new infrastructure required to support the FS design.

- Months 13–24: Pre-Production

  For this period the remainder of the surface material handling infrastructure has been commissioned, but the process plant is still under construction. This allows significant work to progress with setting up the underground in anticipation of the commencement of stoping operations.

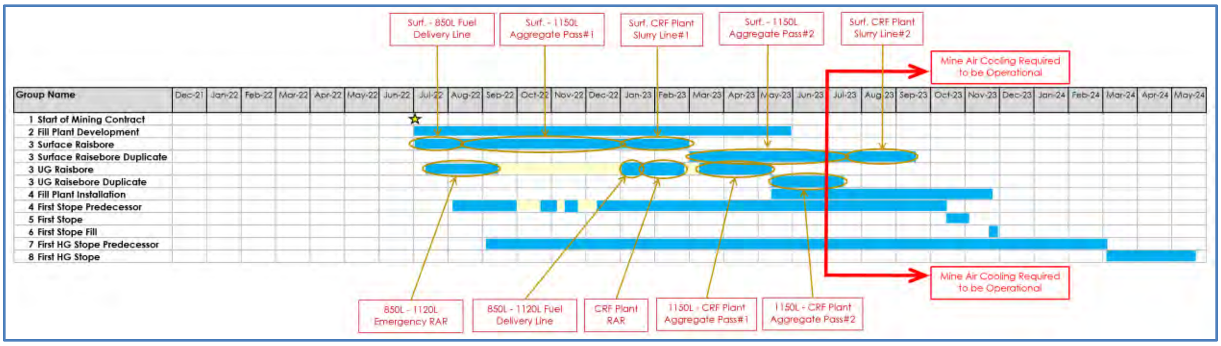
- Months 25+: Production

  With the process plant commissioned, the mine is able to produce at the nameplate production rate of 800 ktpa.

16.4.2 Mine Sequence

Figure 16.55 shows the major milestones for the project.

Figure 16.55 FS Schedule – Major Milestones



OreWin, 2022

Key assumptions:

- Raisebore schedule highlighted as key contributors.
  - The schedule assumes 1 Rig for surface raisebore (RB) holes, and a second rig for underground (UG) RB holes.
  - Assumes 850–1,150 mL Emergency RAR adjacent to flood doors is required ASAP.

- CRF Plant installation and commissioning duration from Golder Fill.
- The schedule assumes Aggregate Pass #2 and Slurry Line #2 can be completed during the CRF plant installation and commissioning.
- Stoping can begin before the CRF plant is commissioned. Based on the assumption that the void will be either stable until the CRF plant is commissioned, or that it can be filled with development waste and then emptied after the CRF plant is commissioned.

Equipment requirements were built up based on the equipment productivity and availability assumptions being applied to the physical quantities of the relevant activity for each piece of equipment.

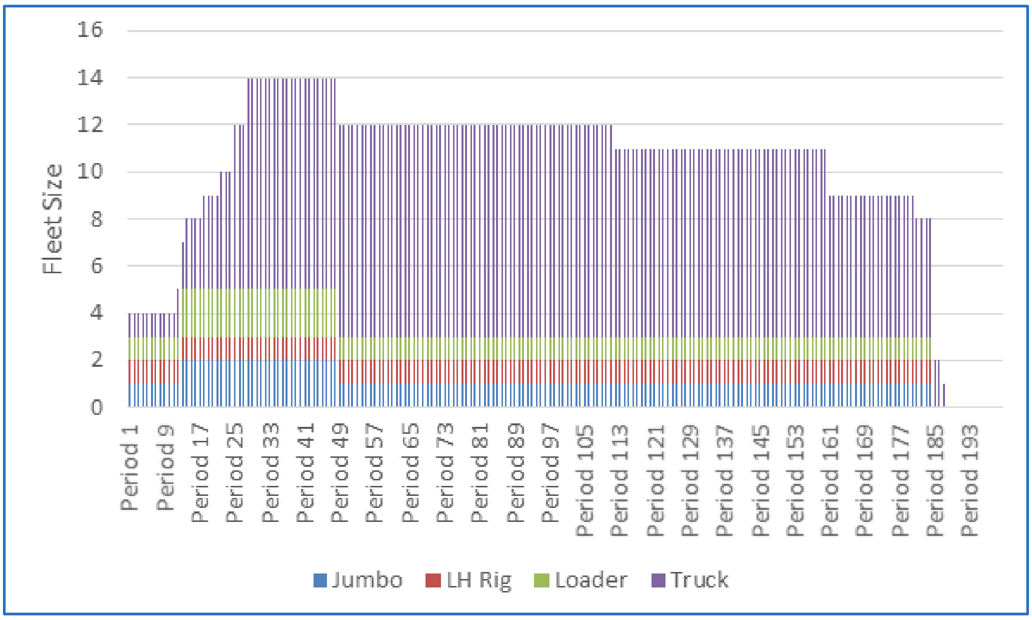
The mobile equipment considered for the report is noted in Table 16.43.

Table 16.43 Major Mobile Equipment Required

Equipment Type		Required
Production Drill	Sandvik DL431-7C	3
Twin Boom Jumbo	Sandvik DD421-60C	3
Large Loader	Sandvik LH517	16
Large Truck	Sandvik TH551	22
Charge Rig	Normet Charmec 605D	8
Raise Drill Rig	Robbins RB56	2
ITC Loader	CAT 930K	3
ITC Loader – Large	CAT 962K	3
Motor Grader	CAT 12K or 140K	2
Utility / Delivery Truck	Normet Multimec	9
Light Vehicle	–	22

The equipment numbers are smoothed where possible to remove fluctuations where the schedule dictates that extra equipment is or is not required for a short period of time. The heavy equipment numbers for the LOM are summarised in Figure 16.56. Where not specified, periods are months.

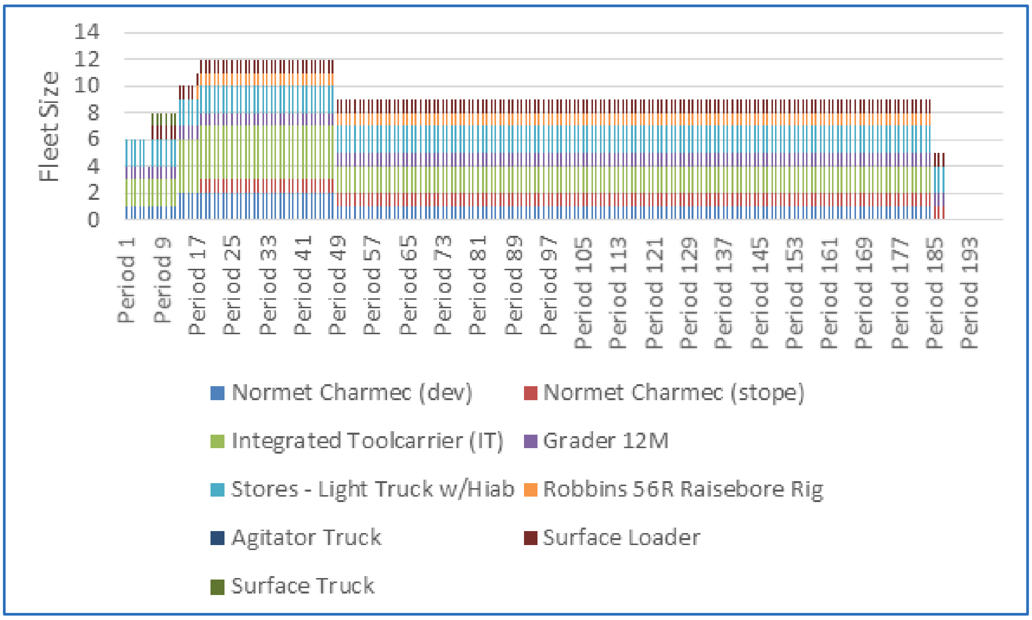
Figure 16.56 Major Equipment Fleet (Owner Operator)



Mining Plus, 2019

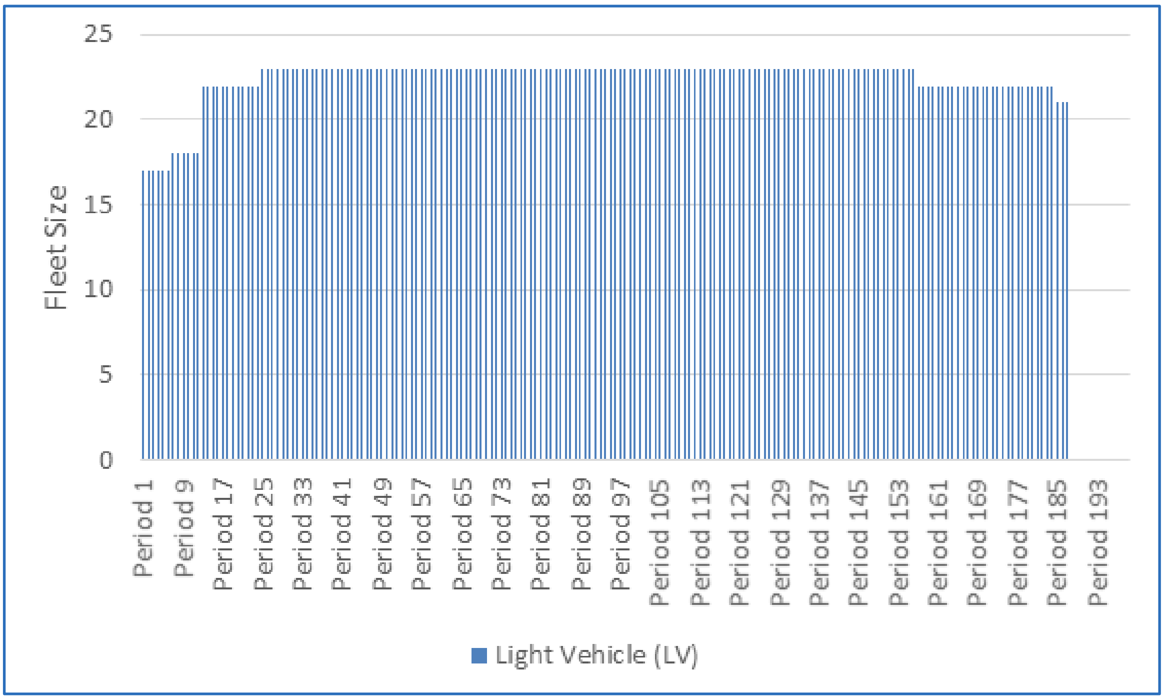
The ancillary equipment fleet schedule is shown in Figure 16.57, with the light vehicle fleet in Figure 16.58.

Figure 16.57 Ancillary Mobile Equipment (Owner Operator)



Mining Plus, 2019

Figure 16.58 Light Vehicle Fleet (Owner Operator)



Mining Plus, 2019

Mine equipment has been selected to fit the profile design sizes and productivity requirements. The major equipment to be used underground is listed below in Table 16.44.

Table 16.44 Major Mobile Equipment

Type	Model
Production Drill	Sandvik DL421-15C
Twin Boom Jumbo	Sandvik DD421-60C
Large Loader	Sandvik LH517
Large Truck	Sandvik TH551
Charge Rig	Normet Charmec 605D
Emulsion Pump	Orica Maxi Loader (up to 3.8T)
Raise Drill Rig	Robbins RB56
ITC Loader	CAT 930K
ITC Loader – Large	CAT 962K
Motor Grader	CAT 12K or 140K
Utility / Delivery Truck	Normet Multimec
Production Drill	Sandvik DL421-15C

16.5 Mine Personnel

The mine will be staffed by a combination of Congolese and expatriate workforce. The expatriate underground roster is a continuous three-panel roster, two weeks on and one week off, day/night shift 12 h rotation. The Congolese roster is a two weeks on, two weeks off, day/night shift 12 h rotation. The underground crews will be a mix of expatriate and national personnel.

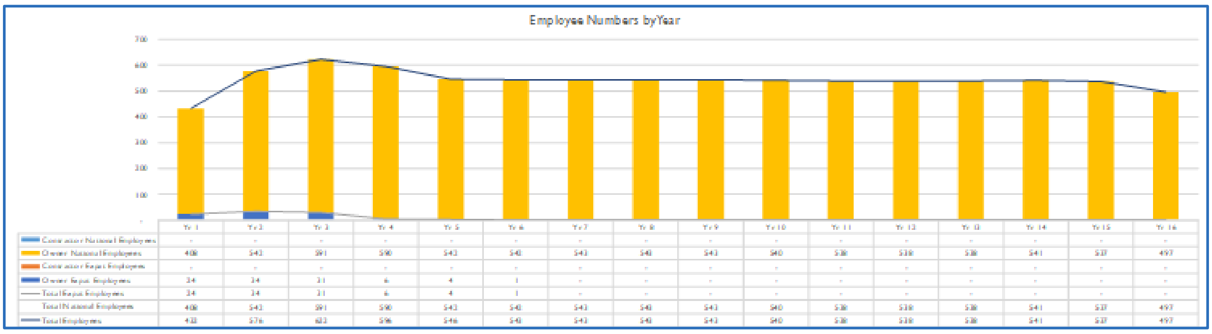
Labour requirements were built up based on first-principles methods and input from other consultants for requirements in specific areas. Operator requirements were built up from first principles based on the operating fleet requirements. Supervisor and maintenance requirements were advised by BOPL and KICO. Manning requirements for other areas were sourced as follows:

- Management and Administration – KICO
- Technical Services – KICO and MP
- Ventilation and Cooling – BBE
- Crushing and Hoisting – M&RC
- Backfill Plant – Golder Fill
- Pumping – M&RC

Where different productivities between the owner-operator model and the contractor model required different fleet sizes, these were reflected in the manning levels. Having determined the required manning levels, the number of personnel required was calculated from the planned roster.

The number of contractor national and expatriate workers per month is fairly consistent throughout the project as shown in Figure 16.59.

Figure 16.59 Workforce Numbers



Mining Plus, 2019

16.6 Mine Consumables

16.6.1 Diesel Consumption

Diesel usage for the mobile fleet was built up from first-principles, using equipment utilisation and fuel burn rates estimated by BOPL.

16.6.2 Ground Support

The ground support requirements for each development heading profile were calculated based on the ground support standards and the number of metres in the schedule for each heading type.

Cable bolts requirements were determined in the mine schedule based on the ground support standards and the number of intersections in the mine.

16.6.3 Drilling Consumables

Drill consumable requirements were calculated based on the development and production drill metres. Drill metres for each metre of development for each profile were calculated on the profile sheet of the cost model.

16.6.4 Explosives

Explosive consumable usage was built up from first principles based on information provided by BOKI.

16.6.5 Electrical Requirements

Electrical power consumption was supplied by other consultants or built up from first principles analysis. The sources were as follows:

- Cooling – BBE
- Ventilation – BBE
- Pumping – M&R
- Crushing and hoisting – M&R
- Other fixed plant – M&R/MP

An indicative power usage for underground has been estimated, based on the underground mobile mining rate by level fleet and the expected power consumption of fixed plant including, ventilation fans, refrigeration plant, underground pumping, underground crusher and hoist, and CRF plant. This is not to supersede the electrical usage study being performed by KICO's consultants but rather as an estimate of the potential usage.

16.7 Mine Ventilation and Cooling Design

Kipushi will be a hot mine that will require refrigeration, it will also be relatively highly mechanised in terms of diesel equipment. The most dominant ventilation design criteria will relate to heat management with the next most dominant being diesel emission management. Ventilation systems that satisfy these criteria will inherently satisfy the other ventilation design criteria. There have been no indications of any extraordinary presence of methane, fibres, radon, silica, etc. and, in the absence of any serious evidence of these contaminants, the design challenge relates primarily to heat and diesel emissions. Thermal issues will dominate the design criteria and ventilation flow rates based on these criteria will, under normal operating conditions, satisfy acceptable gas conditions.

The356356ere356y has a relatively high sulfide content and there will be oxidation reactions which will generate heat, SO_2, and acidic water. Allocations for this heat have been made in the modelling. Regarding possible sulfide dust explosion risks, it is considered that this will be largely mitigated by normal good-practice dust management strategies.

There are no significant gas expectations in this geological area but, if necessary, cover drilling will be carried out to mitigate this possible risk.

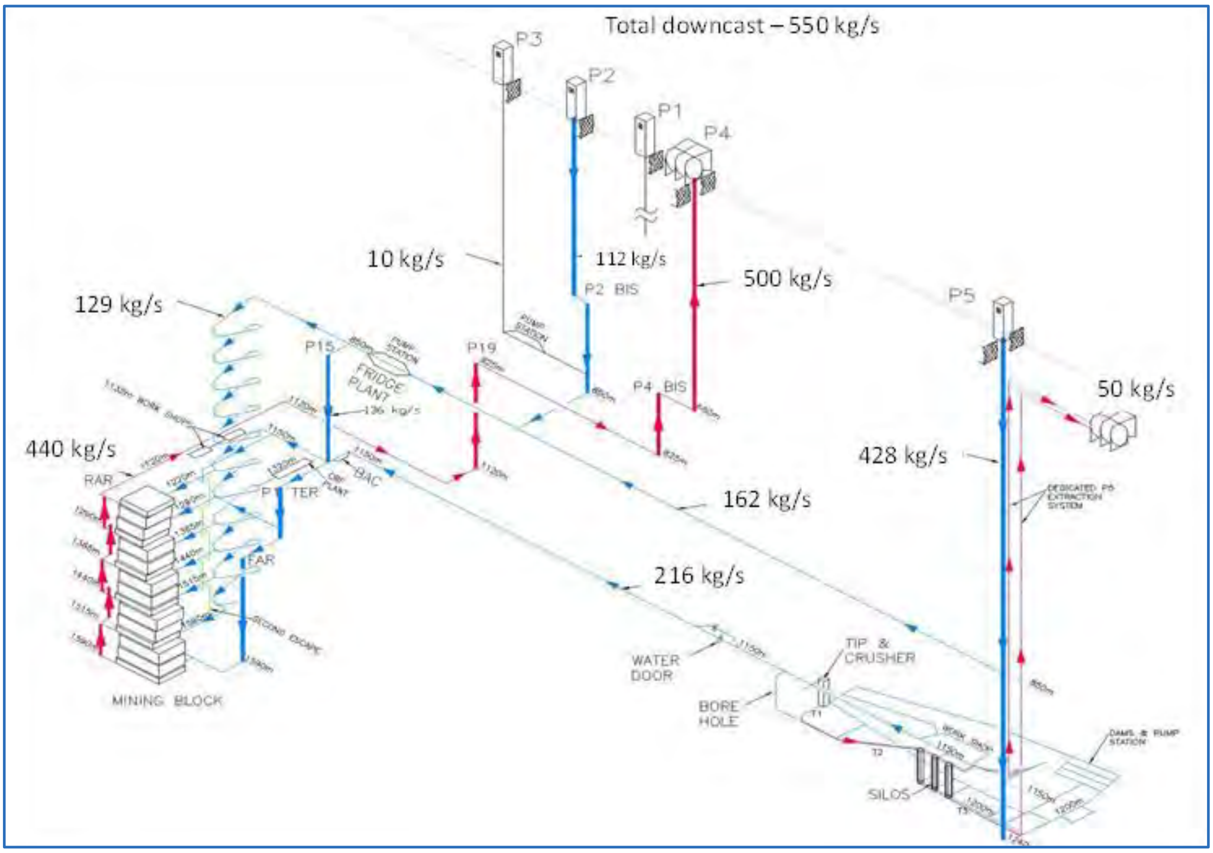
The estimated peak airflow requirement for Kipushi is 550 m^3/s. The airflow requirements are based on meeting a minimum airflow requirement for diesel exhaust dilution rate of 0.06 $m^3/s/kW$ to be circulated.

The overall ventilation specification will be 500 kg/s (+50 kg/s) which benchmarks fairly well against similar operations. The main fan station will have 2 x 2.5 MW centrifugal fan sets and the underground refrigerated air-cooling duty will be 10 MW nominal. These are all reasonable to-be-expected levels of magnitude and there is nothing in this regard that is particularly technically challenging or out-the-ordinary.

Figure 16.60 shows the breakdown of the main flow components with the primary ventilation specification being 550 kg/s.

Table 16.45 is a summary of the primary ventilation shaft flow rates.

Figure 16.60 Primary Airflow



BBE, 2018

Table 16.45 Primary Ventilation – Shaft Flow Rates

Shaft	Flow	Comment
Downcast		
Shaft P5	425 kg/s	Incl. 50 kg/s nom for P5, 1,150 mL/1,200 mL exhaust system
Shaft P2	115 kg/s	
Shaft P3	10 kg/s	
Total Downcast	550 kg/s	
Upcast		
Shaft P4	500 kg/s	
Exhaust ducts in P5	50 kg/s	Nominal
Total Upcast	550 kg/s	

Table 16.46 details how the downcast capacity will be sub-divided in year 2025.

Table 16.46 Downcast Distribution – Year 2025

Location	Flow
Basic Primary Ventilation	332 kg/s
Service Areas (e.g. workshops, etc)	
1,150/1,200 mL, Shaft P5 bottom area	50 kg/s
1,112 mL Sub station	8 kg/s
1,112 mL Diesel bay	10 kg/s
1,112 mL Pump chamber	8 kg/s
1,132 mL Garage	30 kg/s
1,320 mL CRF Plant	25 kg/s
1,150 mL Galleria 19 pump station	5 kg/s
1,170 mL Explosive magazine	10 kg/s
1,182 mL Explosive magazine	10 kg/s
Sub-Total – Service Areas	156 kg/s
General Leakage*	62 kg/s
Total Flow	550 kg/s

*11% of grand total, 19% of basic primary ventilation.

At the reference shaft densities, the volumetric flow rates will be:

- Shaft P4 collar upcast: 550 m^3/s.
- Shaft P5 collar downcast: 550 m^3/s.

These flow rates were selected on the following basis:

- Distribution of adequate cooling from BAC system.
- Underground diesel dilution.
- Maximum carrying capacity at reasonable shaft and main intake air speeds.
- Acceptable air speeds in all underground excavations.

Each of these four criteria is fully satisfied with this primary ventilation specification. For example: the refrigerated air cooling can be provided from a single BAC located at 1,150 mL and delivering ~18°C_{wb} airflow to the system. This is a simple system that is a consequence of the appropriate primary ventilation specification. For example: as a cross-check for diesel dilution: year 2025 will have 4.32 MW 'corrected rated' diesel and at 0.06 m^3/s per kW this gives 259 m^3/s at 1.12 kg/m^3 which is 290 kg/s (which is satisfied by 332 kg/s, Table 16.46).

16.7.1 Airflow Requirements

16.7.1.1 Design Assumptions

The operation will be relatively well mechanised with diesel equipment. The mine will use low sulfur fuel, CF1 low sulfur diesel 50 ppm, as defined by SANS 342:204. Diesel fleet to be used will be minimum Euro Tier II engine specification. The diesel dilution criterion was taken as 0.06 m^3/s per kW rated power being applied at point of use.

16.7.1.2 Airflow Requirements

The Kipushi ventilation design uses a combination of parallel and series ventilation of activities. The primary exhaust is provided on each level. More polluting activities such as production mucking and backfill should be parallel ventilated on the level direct to exhaust. The remaining less polluting development and non-diesel activities can either be parallel ventilated or series ventilated off the decline.

The largest power rated vehicle is the Dump truck at 515 kW and at 0.06 m^3/s per kW these vehicles require minimum ventilation of 31 m^3/s at point of use. The LHDs will require minimum ventilation of 16 m^3/s at point of use. The Dump truck's last point of entry into a development area will be at the last through ventilation position i.e. there will never be a Dump truck and an LHD in the same single heading.

For the parallel ventilation of production mucking activities, sufficient primary airflow must be supplied to each active parallel circuit to cater for the loader and one truck on the level. The other trucks assigned to the loader are assumed to be hauling. There should be no activities scheduled downstream of the production mucking crew on a level when they are working. The airflow rate calculated in this report is designed to cater for a planned production rate of 800 ktpa.

The airflow rates required for parallel ventilated activities are detailed in Table 16.47.

Table 16.47 Parallel Ventilated Level Airflow Requirement

Production Mucking	kW	Airflow (m^3/s)
51 t dump truck	515	30.9
17 t LHD	275	16.5
Total		47

16.7.2 Primary Ventilation Circuit Design

16.7.2.1 Intake Circuit

Air will be downcast from the surface in the vertical, equipped Shafts P5, P2 (and P3 to limited extent). Shaft P5 will be the main downcast shaft off which the flow will split into two main intakes 850 mL and 1,150 mL. These main intakes will be critical and will be run at maximum air speeds <8.0 m/s (note that there will be more traffic on 1,150 mL).

The underground system will include the equipped Shafts P2, Bis and P15 and unequipped Shaft P1 TER together with the Declines. On 850 mL, the air from Shaft P5 will flow to the upper decline and down to 1,150 mL where it will meet with the additional air from Shaft P5, this air downcasts in the Declines and as well Shaft P1 TER. The Declines merge into one at 1,207 mL and Shaft P1 TER ceases as a downcast shaft at 1,285 mL.

The Shaft P2 downcasts from surface to 500 mL from where the airflows to Shaft P2 Bis and then downcasts to 850 mL where it follows two routes, one down equipped Shaft P15 to 1,120 mL where it joins the upper decline and the other down the upper decline to 1,150 mL where it meets the combined airflow from Shaft P5.

16.7.2.2 Return Circuit

The main primary return air strategy is for air to upcast in the RAR/RAG system from the deepest workings to 1,251 mL and then in the RAR/RAG system to 1,120 mL from where it will flow to Shaft P19. From Shaft P19, the air will flow up to 825 mL and then to the bottom of Shaft P4 BIS where it will be upcast to the 650 mL RAW which will transport the air to Shaft P4 where it will be upcast to the surface main fans. The RAR/RAG system will systematically follow mining as production gets deeper. There will be a single exhaust main fan station installed on the surface at Shaft P4 which will have a nominal power rating of 2 x 2.5 MW.

Figure 16.61 shows the intake and return circuits.

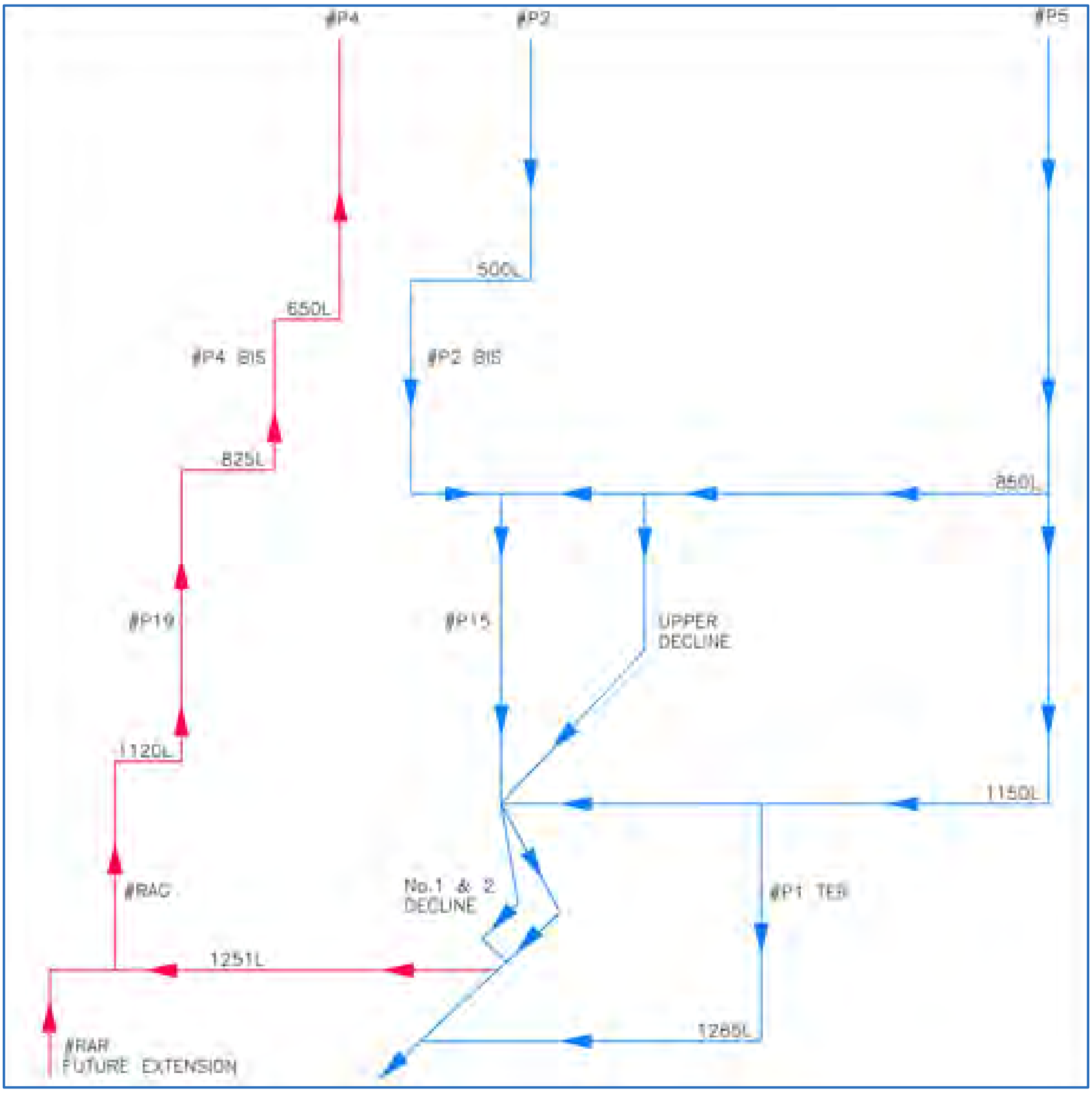
16.7.2.3 Shaft P5 Exhaust Duct System

Figure 16.62 describes the supplementary return system, for the shaft bottom area of Shaft P5, based on two 1 m diameter exhaust ducts that will be installed in Shaft P5 that will exhaust air from the 1,150/1,200 mL Shaft P5 area to surface.

The motivation for this ducted exhaust system was driven by the following:

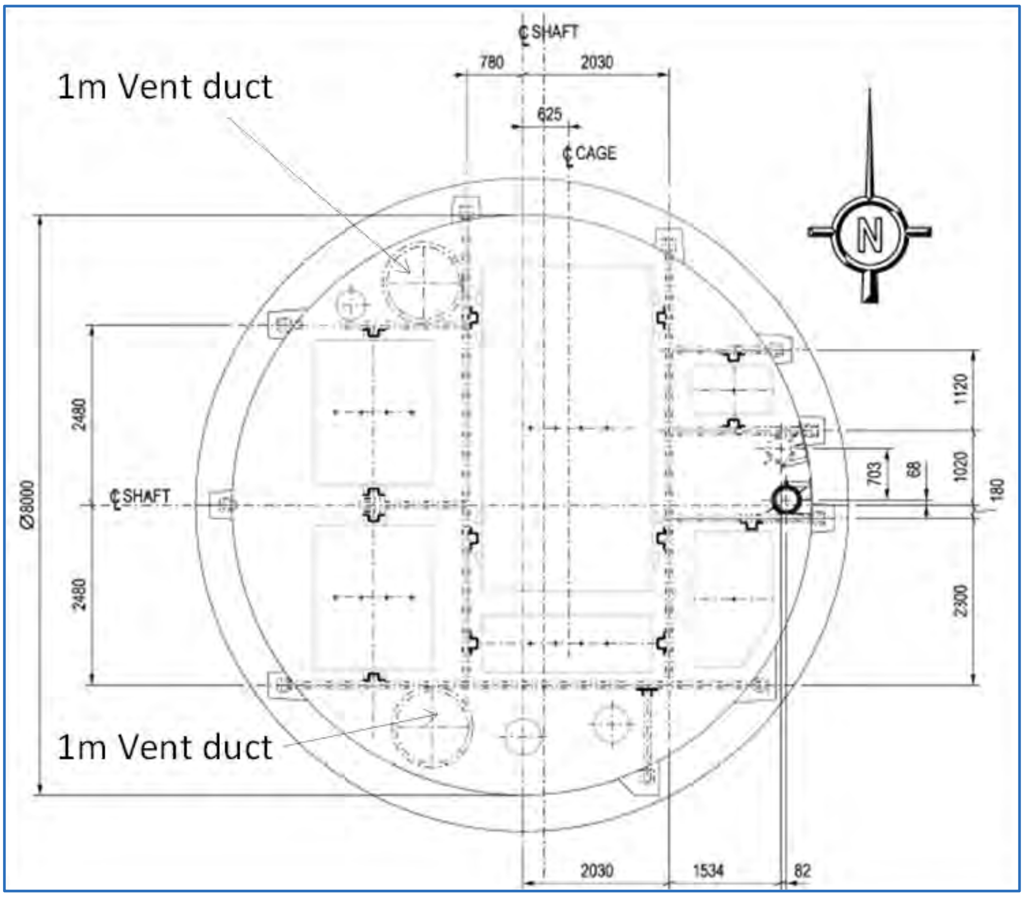
- Prevention of contaminating the entire mine in event of a fire in 1,150/1,200 mL Shaft P5 area.
- Ventilation of main 1,200 mL pump station in event of flooding and water door closure.
- Dust control in the crusher and belt system in the 1,150/1,200 mL Shaft P5 area.

Figure 16.61 Primary Airflow Circuit



BBE, 2018

Figure 16.62 Shaft P5 Ventilation Columns



BBE, 2018

The ventilation columns will provide a total nominal exhaust capacity of 50 kg/s. Thus, the downcast in Shaft P5 will carry this supplementary 50 kg/s which will be dedicated to the shaft bottom area and will be exhausted directly from the 1,150/1,200 mL Shaft P5 area.

For full LOM expectancy, these ventilation columns will be made from 2.5 mm Corten steel. On each duct, a centrifugal fan will be installed in the Shaft P5 sub tunnel general area. Each fan will be sized for 27 m^3/s at 8.0 kPa and will have a rated motor power of 350 kW. KICO have advised that there is space available in the shaft for these columns.

Figure 16.62 shows the preliminary layout of the local exhaust ventilation points.

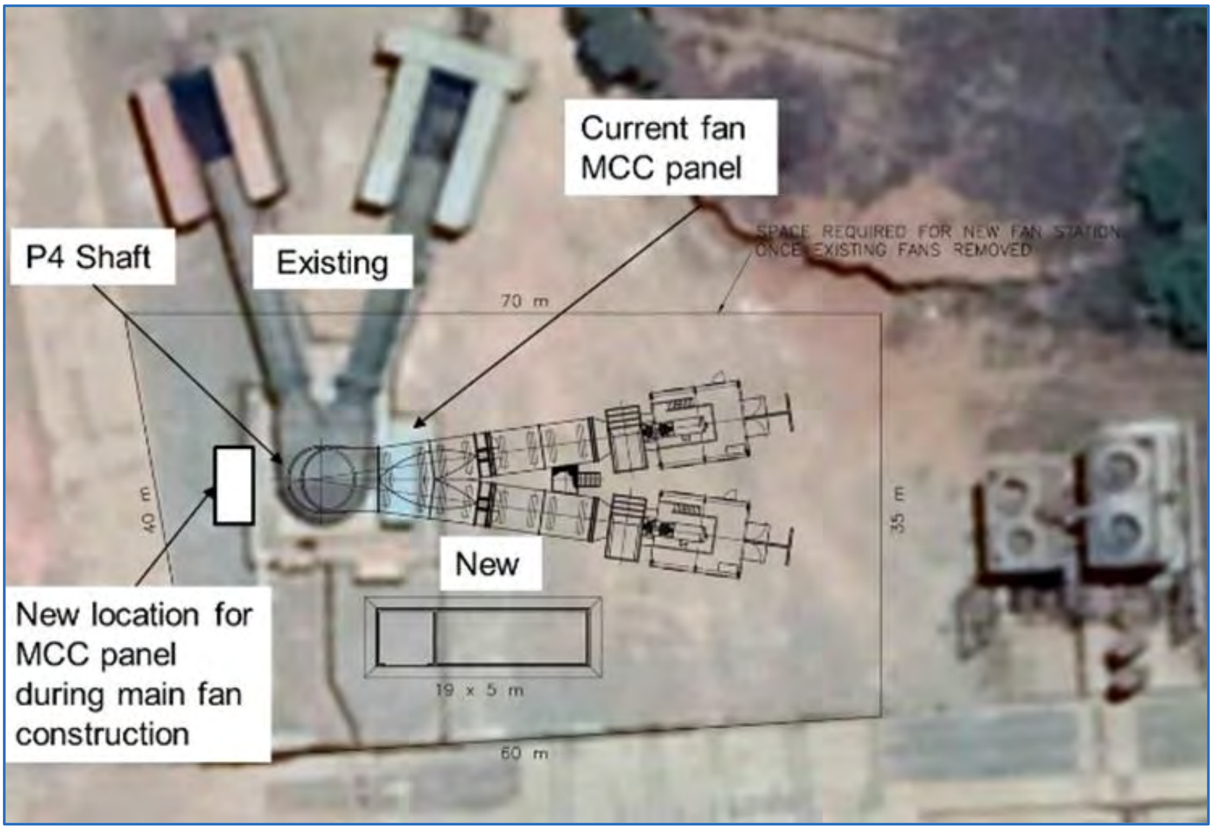
If this extraction system is not favoured the alternative would be to allow for the development of a 2.5 m diameter RBH between 1,150 mL and 850 mL in the proximity of the P5 Shaft area, as well as a review of the emergency response strategy.

16.7.3 Main Fan Stations

16.7.3.1 Shaft P4 Main Fan Station

The new main fan station will be installed on the surface at the existing main exhaust Shaft P4. Two axial fans are currently operating at this location and will need to run until the new fan station can be commissioned. Figure 16.63 illustrates that the new fan station will need to be constructed alongside the operational existing fans and will be orientated in a manner to achieve this.

Figure 16.63 Shaft P4 – Site Layout of New Fan Station



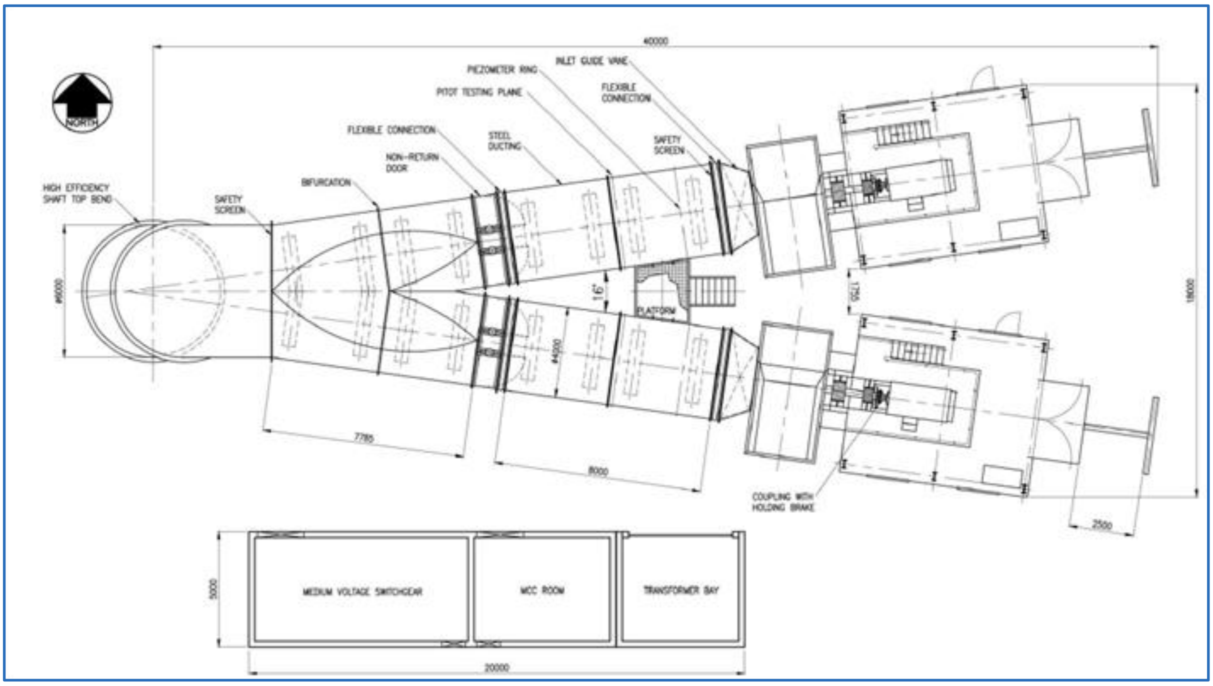
BBE, 2018

The scheduling of the changeover from the old to the new fans will be planned so as to limit the overall downtime. To achieve this, the new fan station will be constructed relatively close to the shaft. The new shaft-top bend will be assembled (in open area next to fan station). The old fans will then be shut down and the old shaft-top bend removed. The new shaft-top bend will be installed and connected to the rest of the new fan station. With careful planning and cold commissioning of the new fans, this changeover should be achieved in a few days.

It is important to note that the motor control centre (MCC) panel for the existing fans will need to be moved to accommodate this construction plan. The MCC panel can be moved to the opposite side of the shaft, see Figure 16.64. It has been indicated that the MCC panel can be moved in a couple of days thus minimising that related downtime.

The new main fan station will comprise a bifurcated drift with two centrifugal fan-motor sets for a total design airflow of 500 kg/s. Figure 16.64 provides a typical general arrangement layout of the fan station (this layout is for descriptive purposes only and will depend on final Vendor details).

Figure 16.64 Shaft P4 – General Arrangement of Main Fan Station



BBE, 2018

Both fans will be required to operate simultaneously from the start-up and will have a minimum life of 15 years. The fans must be capable of operating 24 h/d and 365 d/yr. There will be no connected-in standby unit but strategic spares (such as impeller, shaft, bearings, and motors) will be held on site.

Fan Aerodynamic Performance

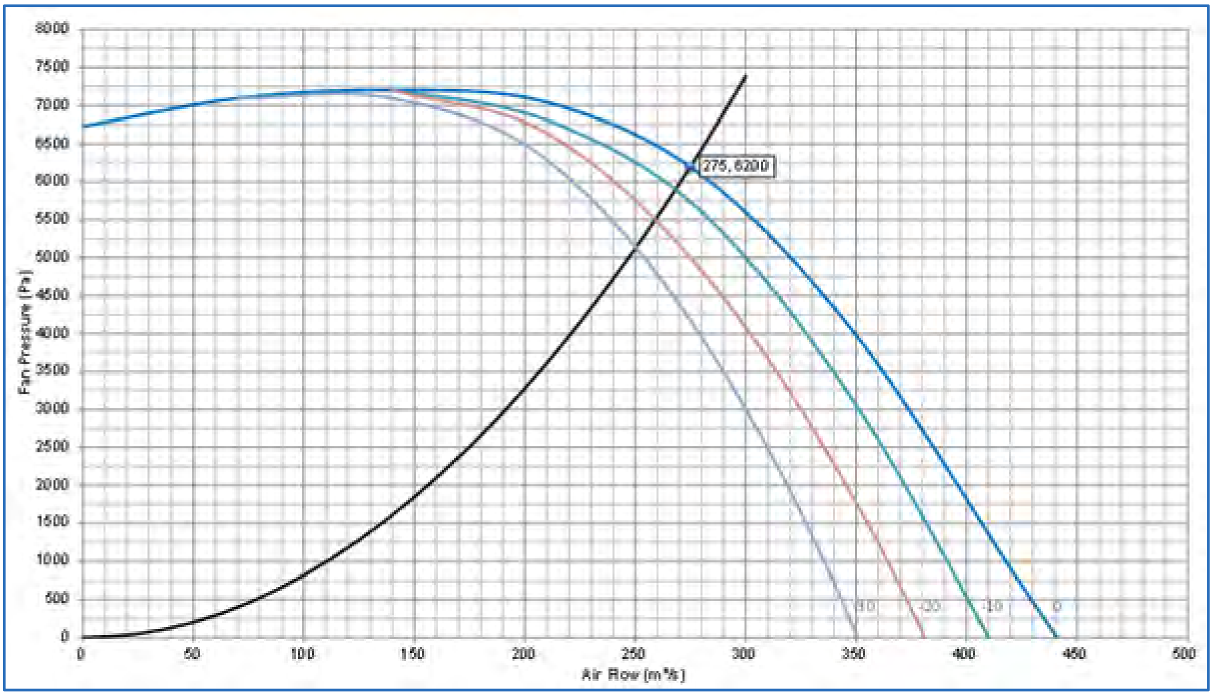
The total fan station design flow will be 500 kg/s (with two identical fans in operation). The nominal process specification for each fan will be as follows:

- Airflow rate: 250 kg/s
- Volumetric flow: 275 m^3/s
- Inlet density to fan: 0.91 kg/ m^3
- Static pressure differential: 6.2 kPa
- Temperature dry/wet-bulb: 22.8/23.0°C
- Fan motor rating (each): 2.5 MW Acceptable air speeds in all underground excavations

The fan pressure stated above is the static pressure measured at the fan inlet and assumes fan drift losses of ~0.35 kPa.

Fan design will be such that an optimum efficiency is achieved at the process point (250 kg/s and 6.2 kPa). The top cover from the operating point to the maximum fan pressure shall not be less than 20% of the maximum fan pressure. Figure 16.65 shows a typical fan curve for this purpose.

Figure 16.65 Main Fan Station Typical Fan Curve



BBE, 2018

16.7.3.2 Shaft P5 Dedicated Extraction Fan Station

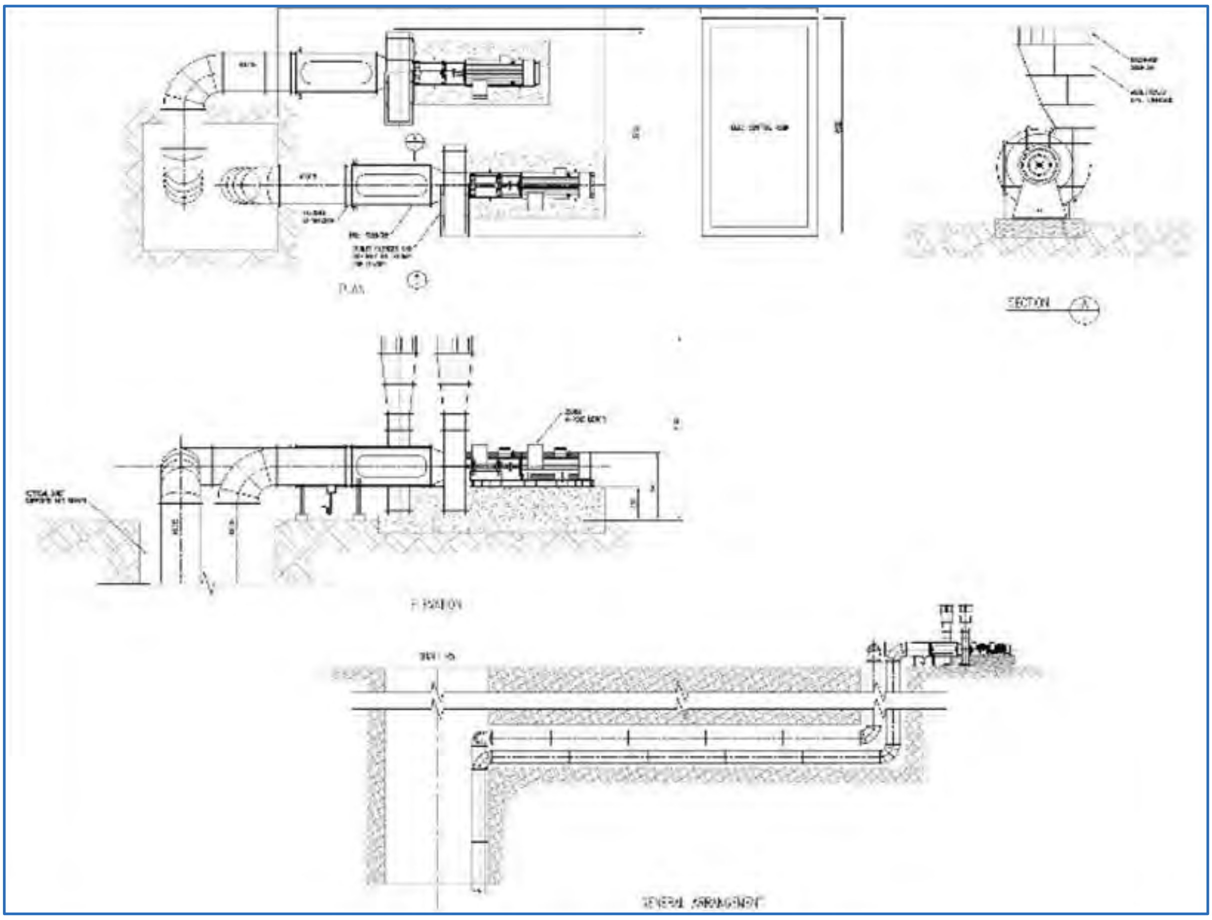
For health and safety purposes as well as best practice, the conveyor belt haulages on 1,150 mL and 1,200 mL will be independently ventilated from the main underground intake airstreams to surface via the Shaft P5 fan station. This system will further provide for dust management along the conveyor belts, where the dust will be captured by hoods and discharged to surface. Figure 16.66 shows where these fans will be situated on the surface at the drift location and will connect to Φ1,015 mm Corten ducting installed in the sub tunnel that connects into the shaft.

Figure 16.67 shows the surface layout of the fan station with the ducting route to Shaft P5.

Figure 16.66 Shaft P5 Extraction System – Site Layout

BBE, 2018

Figure 16.67 Shaft P5 Extraction System – Surface Layout and Ducting



BBE, 2018

16.7.4 Cooling Design

16.7.4.1 Strategy

The determination of the best refrigeration strategy is an iterative procedure between calculating the heat loads and checking the practicalities and sizing for various cooling locations with differing positional efficiency. For simplicity, the iterations are not described here. Rather, the selected strategy is described below, and the heat loads are dealt with in the next section.

In general, for this depth of operation, surface bulk air cooling with surface refrigeration machines would be optimal. This approach was adopted as the early base-case and it was only after it was clear that the existing drift into Shaft P5 was not suitable for cold air feed, that the other options were examined in more detail. For these options examination, the following applies:

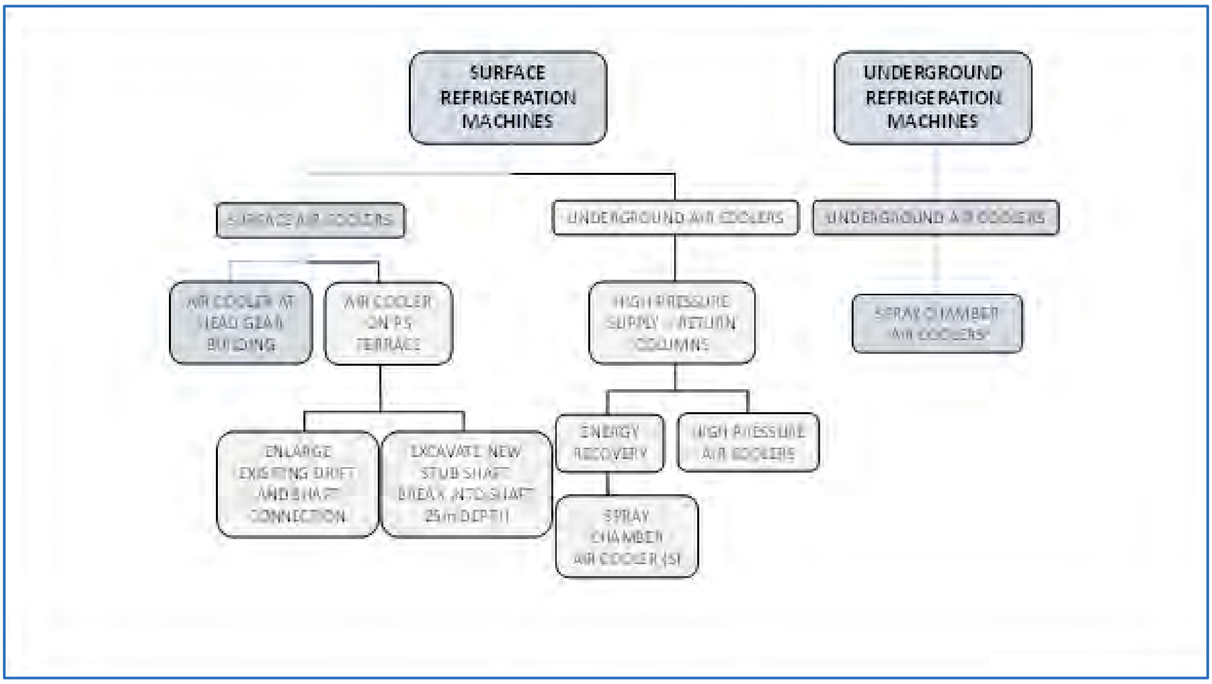
- At ultimate 1,590 mL depth, the nominal air-cooling duty will be ~10 MW – quite substantial.

- Main airway infrastructure in the old mine (Shaft P1, P2, P3 and intake system) is not conducive to applying large cooling capacity.
- The logical location for BACs would be Shaft P5 surface and/or 850 mL and/or 1,150 mL.
- Water pumped out of mine could be used for underground heat rejection – this is a strong differentiator in favour of underground refrigeration machines.
- While the FS ultimate depth is 1,590 mL, the orebody extends to 1,850 mL and there is a probable long-term mine at greater depth. Thus, the possible expansion of the refrigeration system in the long-term is a real consideration.

Figure 16.68 shows the different refrigeration and cooling Options A to F.

Table 16.48 lists the advantages and disadvantages of each of Options A to F.

Figure 16.68 Refrigeration and Air Cooler Options



BBE, 2018

Table 16.48 High-Level Comparison – BAC Options

Item		Features	Advantages	Disadvantages
Surface Fridge Machines	Option A	Enlarge existing drift and shaft break-in	Energy efficient, maintenance simple	High capital for excavation and civils, dangerous excavation work and possible schedule issues
	Option B	Establish new drift and shaft break-in	Energy efficient, maintenance simple	High capital for excavation and civils, dangerous excavation work and possible schedule issues
	Option C	Air cooler at headgear building	Energy efficient, maintenance simple	Possible structural issues at headgear building, cold personnel perception, limited for future expansion
	Option D	Energy recovery system	Air coolers closer to workings, 1,150 mL spray chamber can also do scrubbing	Two air cooler centres, high capital for pipes, les energy efficient, complex equipment underground, maintenance underground.
	Option E	High pressure air coolers	Air coolers closer to workings	Two air cooler centres, high capital for pipes, less energy efficient, complex equipment underground, maintenance underground, fouling of coils
U/G Plant	Option F	Underground plant, heat rejection to dewatering system	Air coolers closer to workings, slightly less duty, 1,150 mL spray chamber can also do scrubbing, suitable for future expansion. Free-up real estate area on surface. No cooling tower plume on surface. Useful waste underground.	More power reticulation to underground, less energy efficient, complex equipment underground, needs underground excavations and maintenance underground.

BBE, 2018

Option A and B were quickly ruled out because of the complexity and cost of excavation and civil work around the Shaft P5 bank. Option D and E were also ruled out quickly because of the need for high pressure shaft pipes and complex equipment underground. The remaining Options C and F were examined in more detail.

The features of Option C (surface option) would be:

- Refrigeration machines in plant house on the surface.
- Heat rejection to surface cooling towers.
- Air cooler off headgear building.

The features of Option F (underground option) would be:

- Refrigeration machines in plant chamber on 850 mL.
- Heat rejection into the watering system and main pump station on 850 mL.
- Air cooler spray chamber on 1,150 mL.

In general, following comparison of Options C and F, the underground option was considered the favoured option, however, there were three issues that needed further discussion:

- Refrigeration compressor power,
- Civils/structures/construction of BAC at headgear vs spray chambers underground, and
- Make-up water consumption.

Refrigeration Compressor Power

Because of better heat rejection, the surface located compressors would run ~4°C cooler than those underground and will be 15–20% more energy-efficient (on average). Based on a nominal cooling duty of 10 MWR, the nominal absorbed compressor power will be 2 MWE, thus the surface compressors will require about 0.35 MWE less electrical power. Based on a power cost of $0.06/kW/h and an annual load factor of 70%, this will relate to about $150kpa in favour of the surface location. In the bigger picture, this is not considered to be a very significant differentiator against the underground location of the refrigeration machines.

Civil / Structures / Construction

Once it was clear that the existing surface drift was not suitable, consideration was given to the possibility of building a surface BAC off the headgear building. The possible concept was to erect a structure on the north side of the headgear above the elevated platform access. However, following a site inspection, it was clear the construction of the air cooler at height will be difficult and costly. It would also be very difficult to seal the building adequately and the practicality of the concept is doubtful. This is a significant differentiator in favour of the underground option.

For the underground option, the following underground nominal excavations will be required:

- BAC spray chamber 1,150 mL: 6 m x 7 m x 80 m long.
- Refrigeration machine chamber: 7 m x 8 m x 40 m long.
- Water storage: 5 m x 5 m x 30 m long.

The new development of all these excavations will be of the order of $0.8 M, however, most of these excavations already exist. An order-of-magnitude estimate for the civils and structures for a surface air cooler, surface plant room, and cooling towers indicates a cost of approximately $3.4M. Thus, even if it was practical to build the air cooler off the headgear (which it is not), it would not be logical from a capex perspective. Again, a strong differentiator in favour of the underground option.

Make-up Water Consumption

The surface option would make use of cooling towers for heat rejection. The towers would require make-up (and blow-down) water. The amount of make-up water would depend on the TDS of the available water. This is unknown at this stage but assuming a nominal 1,000 ppm, the required make-up water flow would be ~400 m^3/d which is fairly significant. It is understood this would probably have to come from a treated water source and this would be costly. No such equivalent consideration applies to the underground machines as they will use once-through water from the 850 mL dewatering system. Again, this is a strong differentiator in favour of the underground option.

16.7.4.2 Selected BAC Option

The underground option was selected as the preferred approach and this was adopted by the project team.

The statement of the refrigeration strategy is as follows: there will be a single large bulk cooler located underground on 1,150 mL near the declines. This air cooler will be served by chilled water from an underground refrigeration plant located on 850 mL. The chilled water pipes between plant and BAC will be carried in Shaft P15.

Regarding the heat rejection from the plant: there will be a condenser water dam established on 850 mL which will collect fissure water reporting to 850 mL. This water will be pumped through the plant condensers and delivered to 860 mL, mixed with other fissure water, and then pumped out of the mine in the dewatering system.

16.7.5 Heat and Ventilation Modelling and Heat-Cooling Balances

In order to determine the refrigeration and ventilation requirements, it is necessary to calculate the overall heat load in the mine (including all the services).

The heat and ventilation modelling were done using the VUMA-3D network package. This software was developed from hot mine applications and is the best possible tool for the purpose of heat, cooling, and ventilation modelling for this mine.

16.7.5.1 Snapshot Years Profiles

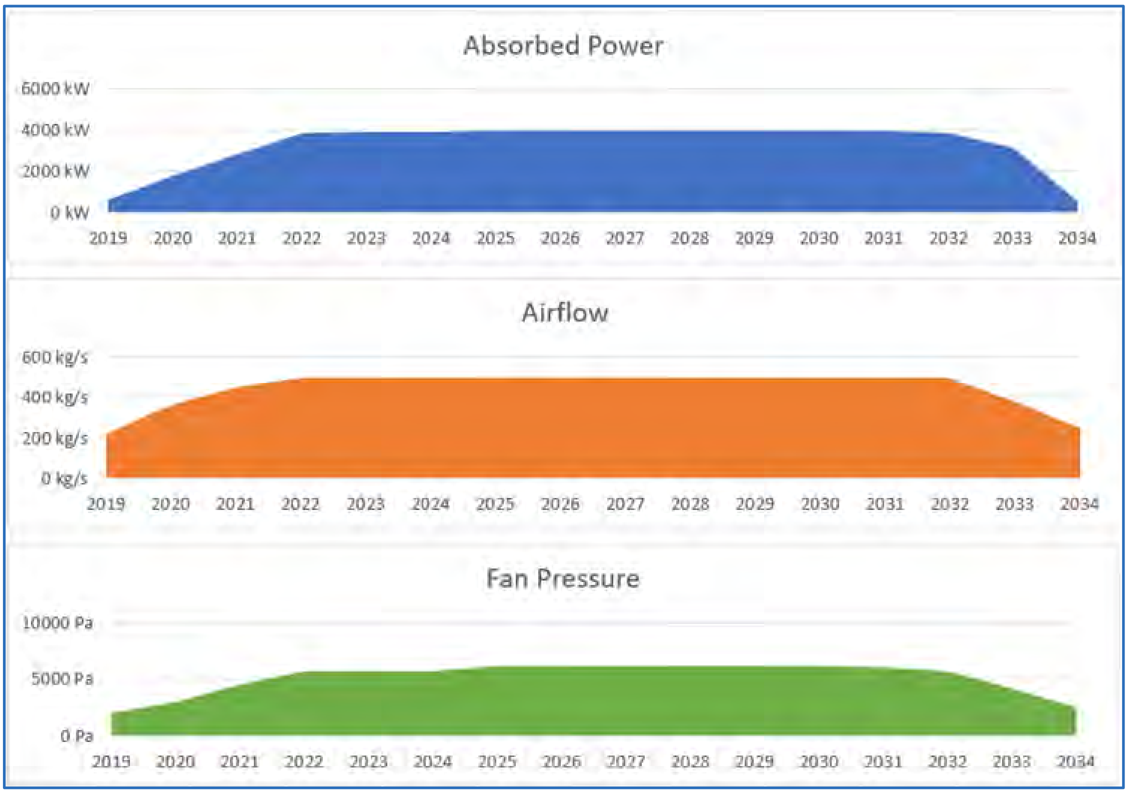
The mining schedule was used to select critical periods in the LOM. The design years examined for the heat, cooling and ventilation were 2020, 2021, 2025, and 2032 and these 'snapshots' were modelled in some detail. In addition and as a sensitivity analysis, a model was examined for an increased ultimate depth of 1,890 m (rather than 1,590 m). The snapshot periods were selected to represent:

- 2020: Mainly development with production on 1,320 mL commencing.
- 2021: Steady-state production achieved with ongoing development.
- 2025: Steady-state production with ongoing development (deeper scenario).
- 2032: Mining commenced in Southern Zinc deposit.

16.7.5.2 Primary Ventilation LOM Profile

The primary ventilation strategy was to maximise ventilation flows in existing infrastructure. Based on this strategy, the primary ventilation flow will build-up from the present 230 kg/s to a maximum of 550 kg/s by the year 2025 and will remain at that value until after 2032 when production starts depleting. Figure 16.69 indicates the VUMA modelling performance profile of the main fan station over the LOM.

Figure 16.69 Required Main Fan Station Performance Over LOM



BBE, 2018

Summary of VUMA modelling as follows:

- Year 2020: Total airflow 360 kg/s Fan pressure 3.0 kPa
- Year 2021: Total airflow 450 kg/s Fan pressure 4.5 kPa
- Year 2025: Total airflow 500 kg/s Fan pressure 6.2 kPa
- Year 2032: Total airflow 500 kg/s Fan pressure 5.7 kPa

The above data led to the selection of a main fan station of 2 x 2.5 MW rated fan-motor sets.

16.7.5.3 Overall Heat Load Components and Heat Modelling Results

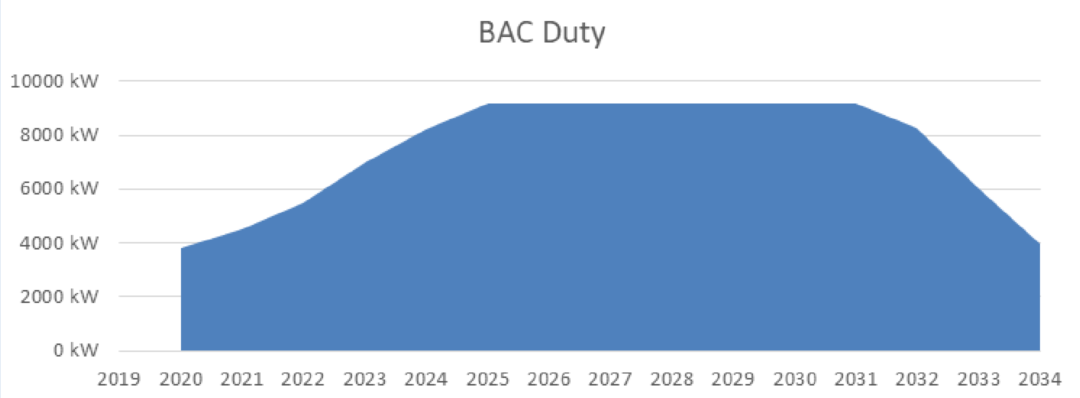
The main feature of the cooling design process has been the fully interactive simulation of the energy balances to determine air temperatures, airflow rates, heat loads, and cooling requirements. VUMA simulation models provide this, and modelling has determined that the total mine heat load is 20.0 MW, for the year 2025 snapshot see Table 16.49. The un-cooled air entering the mine (22°C$_{wb}$) is cooler than the underground reject temperature (28.5°C$_{wb}$) and this contributes a natural inherent cooling capacity of about 10.8 MW. The shortfall of 9.2 MW needs to be met by underground refrigerated air cooling.

Table 16.49 Summary of Heat Loads

Item	Load
Surrounding Rock	1.6 MW
Broken Rock	0.2 MW
Auto-compression	6.8 MW
Diesel Vehicles	5.7 MW
Ground Water	2.4 MW
Auxiliary Fans	0.8 MW
Other Heat Loads	2.5 MW
Total Heat Load	20 MW
Inherent air-cooling power	10.8 MW
Refrigerated air-cooling underground	9.2 MW
Total Heat Sink	20 MW

The heat and cooling balances were determined for each of the snapshot years and then the profile between the snapshots was interpolated, in-line with the mine scheduling. Figure 16.70 shows the indicated LOM profile of the required duty of the BAC.

Figure 16.70 Required Underground BAC Duty Over LOM



BBE, 2018

Summary of VUMA BAC modelling as follows:

- Year 2020 snapshot: 3.8 MW air cooling duty
- Year 2021 snapshot: 4.5 MW air cooling duty
- Year 2025 snapshot: 9.2 MW air cooling duty
- Year 2032 snapshot: 8.2 MW air cooling duty

The full design requirement is therefore needed for 2025 (9.2 MW BAC duty).

16.7.6 Findings and Recommendations

The overall ventilation specification will be 500 kg/s (+50 kg/s) which benchmarks well against similar operations. The main fan station will have 2 x 2.5 MW centrifugal fan sets and the underground refrigerated air-cooling duty will be 10 MW nominal. These are all reasonable to-be-expected levels of magnitude and there is nothing in this regard that is particularly technically challenging or out-the-ordinary.

Due to the frequency of conveyor belt fires and the recent Palabora tragedy, BBE were requested to examine fire scenarios in some detail. There is a conveyor belt system in the Shaft P5 1,150/1,200 mL area which is in the intake pathway and this was examined. In the old mine, two steel ventilation columns exhausted air from 1,150/1,200 mL to the surface and it was decided to re-establish the old historic system as this will allow the conveyor system to be ventilated independently and will allow contaminants from a belt fire to be exhausted to surface directly.

The base case strategy for the refrigeration plant was that it would be located on 850 mL and reject heat into the return mine water on that level. It should be noted that concern has recently been raised as to the capacity and sustainability of pumping this water from 850 mL. Should this concern be realised, the solution would be to re-locate the refrigeration plant on 1,150 mL. Any system design changes that arise from this new information will be addressed post FS in terms of engineering optimisation.

16.8 Hydrogeology

Pumping requirements were based on the 2019 Golder Hydrogeological study. This study has been undertaken to update the initial hydrogeological assessment of the Kipushi Mine completed in 2014. The key objectives of the present study are to aid in quantifying the dewatering requirements for the Kipushi Mining of the Big Zinc Deposit.

16.8.1 Background

Based on the previous studies and the gap closure work undertaken as part of the FS, the conceptual hydrogeological model for the Kipushi Mine has been updated.

16.8.1.1 Historical Mine and Background

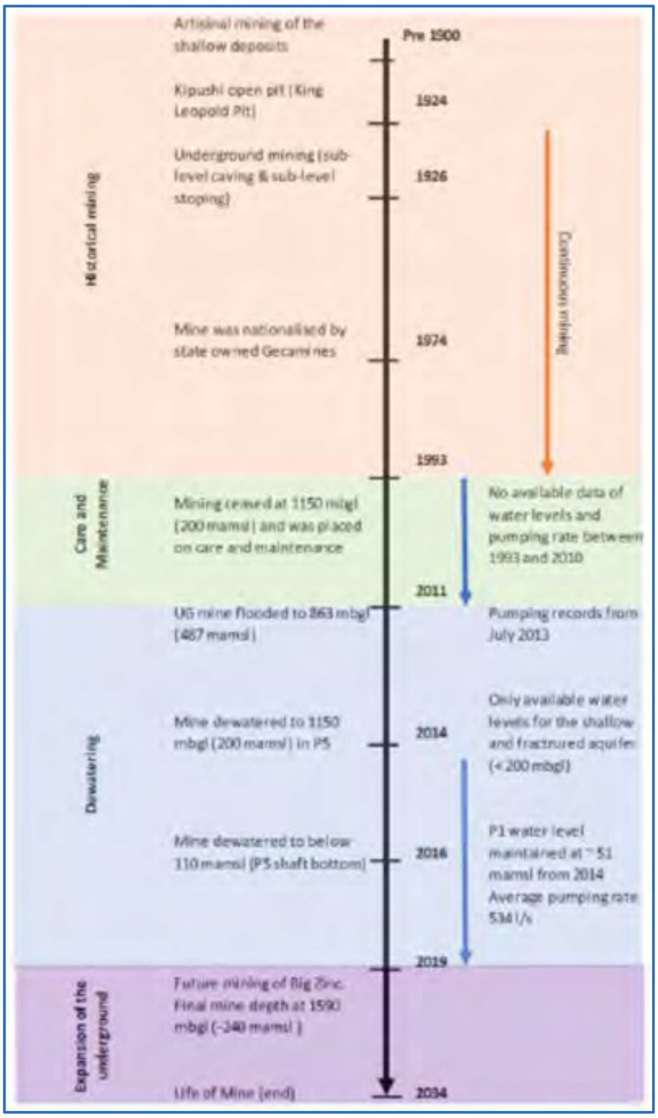
Dewatering initiated in 1929 and continued largely uninterrupted from initiation of mining until 1993. From 1993–2010 the mine was placed on care and maintenance, the pumping requirement to maintain water levels below the 1,150 mL haulage were in the order of 2,400 m^3/h or 666 l/s (Bambele, 2013). In December 2010 pumping was interrupted and by May 2011 the mine flooded to 488 mamsl (832 mL, see Figure 16.71) which represented a 380 m rise in water level.

- The Kipushi mine has targeted the north-westerly dipping Kipushi Fault zone from near surface to 20 mamsl (1,330 mL). The future mining of the Big Zinc deposit will be mined over a 15-year period and will reach a depth of –240 mamsl (1,590 mL).
- The Kipushi fault zone cross cuts a regional anticline. The hanging wall consists of axial breccia while the footwall comprises Kakontwe dolomites. The units are regarded regionally to be water bearing.
- The mine is currently (February 2018) required to pump approximately 1,980 m^3/h (550 l/s) to maintain dry conditions within the mine.

16.8.1.2 Current Water Levels and Cone of Depression

- Dewatering of the deep underground mine has resulted in partial dewatering but not complete dewatering of the upper aquifer. The partial dewatering is consequence of the reduction in conductivity with depth reduces the connectivity of the mine with the upper aquifer.
- The drawdown in the upper aquifer occurs in the Kakontwe dolomites and the axial breccia primarily. Based on known water levels, the cone of depression does not develop north of the mine. The cone of depression is limited as a consequence of the low conductivity Nguba shales and the Petit Conglomerate formation. Consequently, water levels beneath Kipushi town (Kiuobo dolomitic sandstones) have not historically been affected by dewatering and are not anticipated to be affected by dewatering in the future.

Figure 16.71 Kipushi Mine Dewatering Timeline



Golder, 2019

16.8.1.3 Aquifer Properties

Aquifer testing results were highly variable (typical of fractured rock environment). The values obtained for the upper aquifer (above 150 mbgl). The results of the tested values are summarised below:

- Kiubu Sandstones (Ku2.2): The conductivity from testing was 0.15 m/d.
- Petit conglomerate (Ku1.1): was not the target by a borehole.
- Monwezi shales (Ng2.2): This zone together with the adjacent diamictite are interpreted to be of low permeability and limits the extent of the cone of depression. The tested conductivity is 0.023 m/d.

- Katete shales (Ng2.1): The tested conductivity for this zone is 0.025 m/d.
- Kakontwe dolomite (Ng1.3): The tested values of this zone 0.017 m/d.
- Grand Conglomerate (Ng1.1): The tested value was 0.022 m/d.
- Axial breccia: The testing yielded values of 7 x 10–3 m/d, which is lower than expected.
- Mwayasha subgroup (R4): The tested value for this unit was 6 x 10–3 m/d.
- Dipeta subgroup (R3): The very high (K >10 m/d) tested value and was taken to represent a fault or a fracture.

Table 16.50 shows the measured Hydraulic Conductivity (m/d) from Literature and used in Hydrogeological Model.

16.8.2 Inflows

Within the mine, the testing and the interpretations of the site visit indicated that inflows gradually decreased with depth and consequently the transmissivity is expected to decrease with depth. The Pressure Shut In Tests at the 1,180 mL resulted in hydraulic conductivity values that ranged between 0.03–0.09 m/d.

Table 16.50 Hydraulic Conductivity (m/d) measured, from Literature and used in Hydrogeological Model

Formation	Code	Measured	Expected low from literature*	Expected high from literature*	Model deepest layer	Model upper layer	Comment
Kiubu Sandstone	Ku2.2	1.50E-01	6.80E-03	1.87E-01	6.80E-03	1.87E-01	Within Expected Range
Petit Conglomerate	Ku1.1		5.23E-06	5.00E-03	1.00E-04	1.00E-02	
Monwezi Shales	Ng2.2	2.30E-02	7.20E-06	2.60E-02	2.60E-03	7.20E-02	
Katete Shales	Ng2.1	2.50E-02	7.20E-06	2.60E-02	5.00E-04	1.40E-02	
Kakontwe Dolomite	Ng1.3	1.70E-02	2.62E-03	7.00E-01	2.60E-02	7.20E-01	
Grand Conglomerate	Ng1.1	2.20E-02	5.23E-06	5.00E-03	2.60E-02	7.20E-01	
Axial Breccia	AB	7.00E-03	2.62E-03	7.00E-01	2.60E-02	7.20E-01	
Mwasyasha Shales	R4	6.00E-03	7.20E-06	2.60E-02	1.00E-04	1.00E-02	
Dipeta Shales	R3	>10	7.20E-06	2.60E-02	1.00E-04	1.00E-02	

*Freeze and Cherry (1979); Heath (1983); Spitz and Moreno (1996).

16.8.2.1 Recharge and Recirculation

- Natural recharge on the shales and diamictite is likely to be in the order of 0.5–18.75 mmpa based on model calibration. Similarly, recharge on the Sandstone was calibrated to be in the order of 31.5 mmpa and recharge on the Axial Breccia was calibrated to be in the order of 37.5 mmpa.
- Enhanced recharge on the historic open pit and as a consequence of artisanal mining was concluded to have a negligible effect on the total dewatering volumes currently pumped from the mine.
- From the study results, it was concluded that recirculation from the northern unlined channel is in the order of 8–12% (i.e. 70 l/s). Despite the tracer test results being inconclusive, it was inferred from the streamflow measurements, the known groundwater levels, and geological mapping that these losses occur within the first 500 m of the channel. After this point negligible stream loss occurs.

16.8.2.2 Sources of Ingress to the Underground Mine

- The water entering the mine is enriched in sulfate and chloride and isotopically distinct from the shallow aquifer. These findings together with review of the artificial tracer tests has concluded that the interaction between the upper aquifer (recently recharged zone) and the mine is not rapid as previously postulated. It follows that fault zones surfacing in the upper aquifers do not rapidly transmit water to the mine at depth.
- Ingress into the mine is controlled in part by fractures. Mapping of faults within the decline indicated most of the water bearing fractures are perpendicular with the Kipushi fault and hence dip east–south-east. The underground site visit observations found that the drilling of these holes created preferential flow paths for groundwater ingress into the mine.

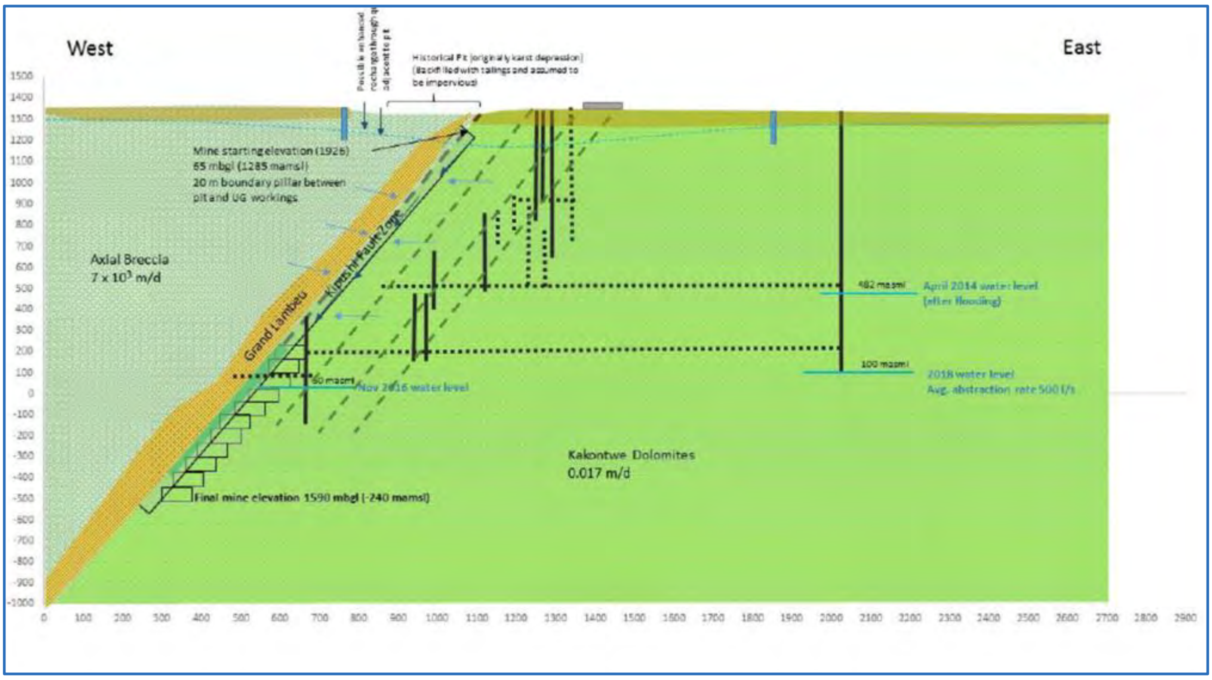
16.8.3 Hydrogeological Model

Based on the previous studies and the gap closure work undertaken as part of the FS, the conceptual hydrogeological model for the Kipushi Mine has been updated and is summarised below and depicted in Figure 16.72.

16.8.3.1 Previous Hydrogeological Model

The previous hydrogeological baseline assessment involved a field programme comprising the drilling of 24 boreholes and the development of an initial numerical model to aid in simulating the effects of mine dewatering.

Figure 16.72 Conceptual Model



Golder, 2019

- The 2014 model did not incorporate the future mine plans and the indicative dewatering assumed the mine was to remain at 200 mamsl.
- The recommendation of the previous model was to undertake aquifer testing to obtain site specific hydrogeological parameters and incorporate the mine plan proposed for exploitation of the Big Zinc deposit.

16.8.3.2 Re-Calibration of the Numerical Model

The model has been updated in order to include the mine plan proposed for the Big Zinc Deposit. The plan has a final mine depth of –240 mamsl. This required the existing model to be re-meshed and the depth (thickness) of the model increased to allow for incorporation of the mine plan. Model parameters were adjusted where necessary in line with measured hydraulic conductivity values and calibration thereof.

The key conclusions drawn from the recalibration of the numerical model are:

- Significant data gaps exist for the study area despite the legacy of mining at Kipushi. The key data gaps include:
  - No structural data is available beyond the mineralisation zones. Structural features are conceptualised to be an important mechanism via which groundwater from the shallow aquifer enters the mine workings. Due to uncertainty regarding the position and behaviour of these structures, the representative elementary volume (REV) approach was followed in estimating hydraulic conductivity values.

  - Water level data is available for 2014/2015 proximal to the mine although no water levels are available beyond a 1 km radius from the mining area. As such the steady state flow model is assumed to behave correctly.
  - Mining has been underway since 1926. However, dewatering volumes are only available from 2013 to present and water levels within the underground workings from 2010 to present.
  - River flow data is not monitored and thus an indication of recharge to the system cannot be made and remains an uncertainty.
- The previous report (Golder Associates, 2017) describes the model sensitivity to variations in parameters.
- The recommendation of the previous model was to undertake aquifer testing to obtain site specific hydrogeological parameters and incorporate the mine plan proposed for exploitation of the Big Zinc deposit.

16.8.3.3 Future Mine Dewatering – Big Zinc and Southern Zinc Deposits

- Following completion of calibration, the best combination of recharge, hydraulic conductivity, and storage parameters determined from the sensitivity analysis was carried through to the predictive scenario modelling phase.
- The predictive modelling indicated mine dewatering is expected to peak in 2023 with the maximum expected inflows approximately 830 l/s. This is considered to be a conservative estimate, therefore not likely to be exceeded. However, if a 10% uncertainty is included, the inflows can reach 795 l/s.
- The proposed mine water reticulation system has a design capacity to pump 833 l/s to surface. Based on the simulated inflows, no additional pumping capacity is required.
- Although the simulated values are below the current pumping capacity, it is recommended that the inflow rates be carefully monitored and compared with the predicted inflows. If a discrepancy is observed, the model should be updated and re-calibrated to see if provision for additional pumping will be required.
- At present there is no knowledge of highly permeable zones associated with faults or fractures in the mining area. There is a possibility that such a high permeable zone may be encountered during mining which will result in increased flow into the mine and additional pumping capacity may be required. This is a potential risk that will need to be considered, but without any data indicating a high permeable zone the risk is perceived to be low.
- The cone of depression is expected to propagate east along the Kakontwe Dolomite and Axial Breccia. The development of the cone is limited to the north by the Petit Conglomerates. Thus, the resulting cone of depression is expected to be elongated in a west–east direction following the trend of the Kipushi valley. The addition drawdown from 2019 to 2034 is in the order of 38 m.
  - Although the Petit Conglomerates formation acts as a barrier, the water users to the north of the mine may be impacted by continued drawdown.

  - The existing Kipushi well field is located on the Axial Breccia. Limited water levels could be found for these wells (P1 possibly has an 82 mbgl water level in 2014). However, based on the cone of depression simulated it is possible that these wells are already impacted by mine dewatering.
- The simulation of dewatering of the future mining area indicated that the strata overlying the mine remains saturated through time and a steep cone of depression is expected adjacent to the mining areas throughout the LOM. Based on the simulated pore pressure distributions, it is expected that high pore pressure will persist in the rock faces adjacent to the mining voids which may be a potential risk.

16.8.3.4 Water Level Recovery After Mine Closure

The model showed that water levels will take more than 100 years to recover to steady-state conditions. After 100 years, water levels are 3 m below steady-state water levels for the backfilled scenario and 96 m below steady state water levels for the open void scenario.

16.8.3.5 Solute Transport

The extent of the simulated sulfate plume is larger during the operational and transitional phases, but thereafter, the plume size decreases due to the smaller influx (and lower concentration) after the transitional period when the tailings are dry. However, there is still an influx and the concentration continue to increase until at least 100 years after closure. At some point in later future, all the sulfate will be leached out of the TSF and the concentration should then gradually start decreasing, but this will not happen in the simulation period until 100 years after closure.

The plume extends further for longer in the open void scenario, but the concentrations are much lower than in the backfilled scenario. This can be explained by looking at the groundwater level gradient in the TSF area. The water level is very flat for the backfilled scenario, but for the open void scenario, there is a gradient from the TSF to the west. Since there is no gradient in the backfilled scenario, the water is almost stagnant, and the concentration builds up under the TSF. For the open void scenario, the gradient allows for water movement away from the TSF as well as dilution.

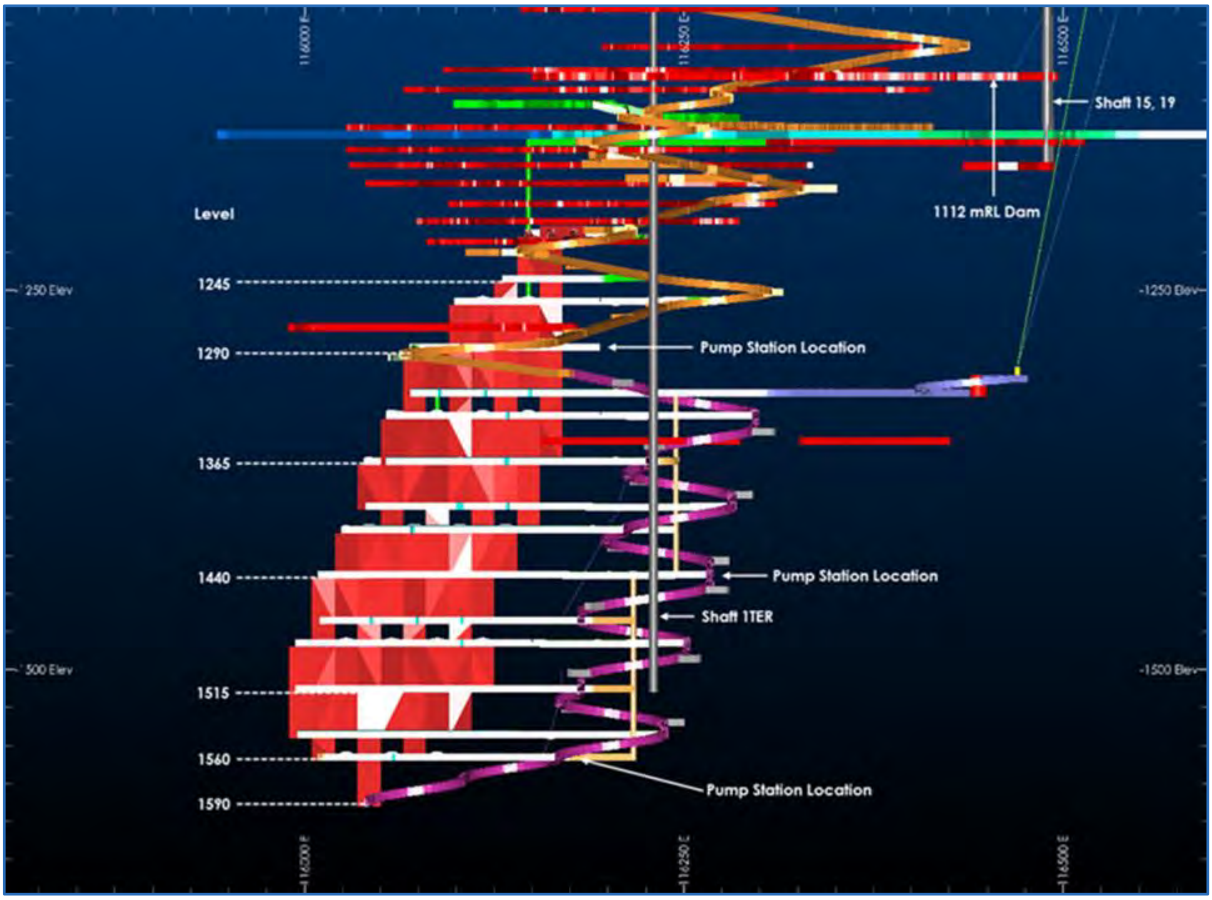
16.8.4 Dewatering System Design

At the mining depth of 1,560 mL, the maximum predicted inflow occurs at 2,863 m^3/h. The dam on the 1,112 mL is the closest location to the proposed Kipushi 2022 FS designs that, in turn, dewaters to the surface. Therefore, at the maximum mining depth and inflow a dewatering pumping system is required that is capable of moving approximately 3,000 m^3/h up to 480 m of head. Figure 16.73 shows the proposed dewatering levels over the LOM.

As indicated in Table 16.51 the future mine simulations indicate the inflows to the mine expected to peak at 722 l/s. Based on the reticulation diagrams the proposed dewatering pumping network can handle a dewatering rate of 3,000 m³/h (or 833 l/s). Thus, the simulated inflows are within the proposed pumping capability of the mine. However, due to the uncertainty with regards to aquifer parameters (conductivity and storage), a 10% range on the base case inflow scenario has been considered. Thus, the upper limit of potential inflows, possibly up to 795 l/s, will be within the current pumping capacity to dewater the underground mine. Although these values are below the current pumping capacity, it is recommended that the inflow rates be carefully monitored and compared with the predicted inflows. If a discrepancy is observed, the model should be updated and re-calibrated to see if provision for additional pumping will be required.

At present, there is no knowledge of highly permeable zones associated with faults or fractures in the mining area. There is a possibility that such a high permeable zone may be encountered during mining which will result in increased flow into the mine and additional pumping capacity may be required. This is a risk that will need to be considered, but without any data indicating a high permeable zone the risk is perceived to be low.

Figure 16.73 Proposed Dewatering Levels over LOM



OreWin, 2017

Figure 16.74 shows the updated water handling process flow diagram, showing the proposed workings and the pumping system to the 1,112 mL dam. It must be noted that phase one, phase two and phase three dewatering pumping stations occur separately as mining progresses.

Dewatering pumps are initially located on the 1,290 mL and are moved to the 1,440 mL station as mining progresses past the 1,440 mL. Once mining has progressed past the 1,560 mL dewatering pumps are moved from the 1,440 mL station to the 1,560 mL station. Additional pumps are purchased as required, as inflow increases with depth. Dewatering pumps will remain on the 1,560 mL station for the LOM. Levels between the pumping stations will either feed the dewatering stations through the use of submersible pumps, from sumps, or by gravity.

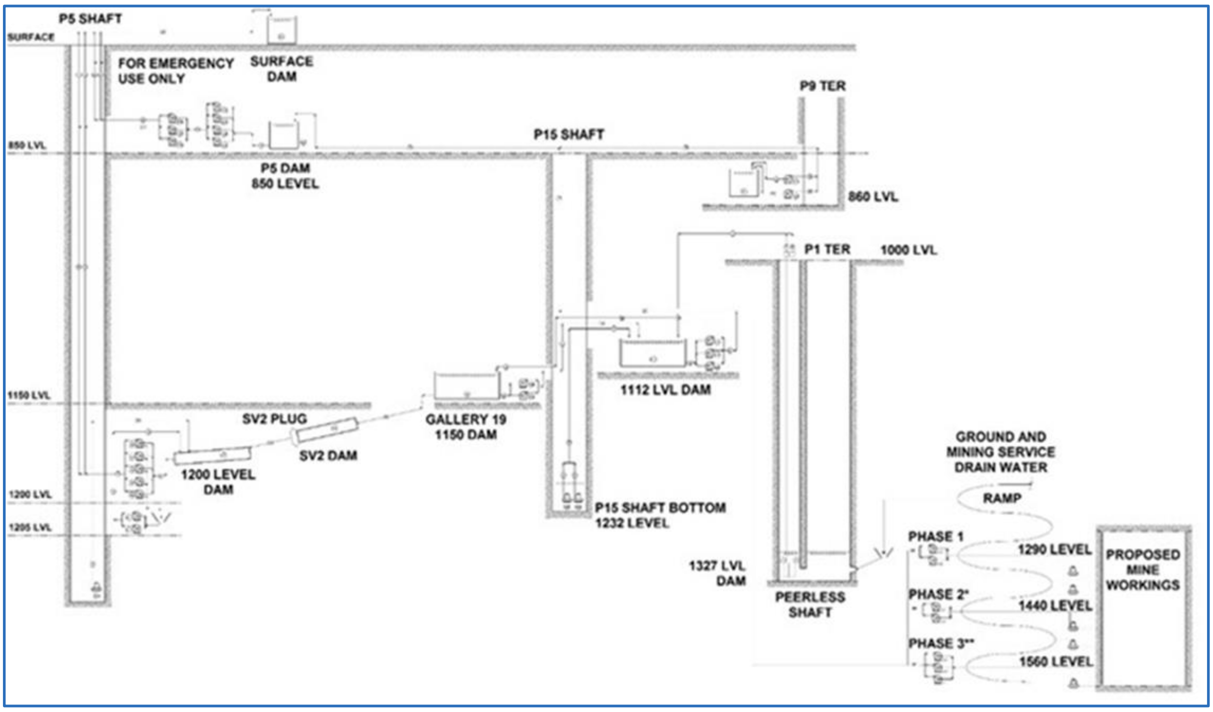
Table 16.51 Simulated Inflow Rates in l/s (average value per year)

Year	Deepest Mining Level	Above 850 mL	850 mL – 1,150 mL	Below 1,150 mL	Lower Range (Total –10%)	Total (sum of 3 levels)	Upper Range (Total +10%)
2021	1,335	270	219	150	575	639	702
2022	1,395	266	187	216	602	669	735
2023	1,560	260	179	248	619	687	756
2024	1,560	254	171	289	643	714	786
2025	1,560	247	163	312	650	722	795
2026	1,560	240	158	306	634	704	775
2027	1,560	236	156	304	626	696	765
2028	1,560	231	155	303	620	689	758
2029	1,560	228	154	302	615	684	752
2030	1,560	225	153	301	611	679	747
2031	1,590	223	153	339	643	714	786
2032	1,590	219	151	295	599	666	732
2033	1,590	217	151	298	600	667	733
2034	1,590	216	151	298	599	665	732
2035	1,590	214	151	298	597	663	730
2036	1,590	213	151	311	607	674	742

During the LOM, multiple active mining levels are operational, with up to four stopes or pillars being extracted simultaneously. Submersible pumps located on the active mining levels will pump to dewatering stations positioned off the decline. Dewatering pump stations will initially be located on the 1,290 mL, then as mining progresses on the 1,440 mL and lastly the 1,560 mL.

Based on the pump specifications, two centrifugal dewatering pumps would be required on the 1,290 mL to dewater to the 1,112 mL dam. When mining reached the 1,440 mL the pumping station would be moved to this level and again, two pumps would be required to feed the 1,112 mL dam. Finally, when mining reaches its full depth at the 1,590 mL, the pumping station would be moved to the 1,560 mL, where three pumps would be required to meet quantity and head requirements to feed the 1,112 mL dam. Table 16.52 shows the specifications of a centrifugal dewatering pump which would meet the demand.

Figure 16.74 Updated Water Handling Process Flow Diagram



OreWin, 2017

Table 16.52 Pump Specifications

Item	Units	Value
Power Consumption	kW	2,000
Power Frequency	Hz	50
Capacity	m³/h	1,000
Efficiency	%	82.8
RPM	rpm	1,490
Head	m	480

With stopes and pillars being mined simultaneously on multiple levels at a time, water inflow from exposed faces must be managed via the use of submersible pumps, located in sumps feeding the decline dewatering stations. Gravity feed will also be employed where convenient and depending on the level being mined. Based on scheduling and varying active mining levels, it was calculated that eight pumps would be required for LOM.

16.8.5 Recommendations and Way Forward

- It is recommended that additional boreholes should be drilled. The purpose of the boreholes is:
  - To verify the simulated water level drawdown north of the Petite Conglomerate.
  - To check the influence of mining on the well field.
  - To check the water quality downstream of the TSF in comparison with upstream water quality.
- The model is a water management tool that should be updated regularly (every two years) to incorporate ongoing monitoring data and to adjust the model as additional information becomes available.
- The model should be updated sooner if any additional inflows or other changes are observed during mining. Immediate action should be taken as soon as changes (i.e. should high permeable zones be encountered during mining) are observed.
- It is recommended that an unsaturated flow model should be constructed to investigate in more detail and to include the effect of evaporation. For this simulation, evaporation was included in the sense that a lower flux into the tailings was specified. An unsaturated flow model will give a specified flux value with higher certainty.

Water quality is generally good, but poor water quality was observed in isolated places within the underground mine. Water quality issues will be revisited and included in the post-closure scenarios and mass transport models.

16.9 Production Plan

The following general planning criteria were applied to determine priorities for initial production:

- Extraction of primary SLOS stopes before secondary stopes as Figure 16.44.
- Mining of SLOS extraction level before sublevel.
- Mining of the Pillars only once sublevel below and extraction level above are mined.
- Highest Grade.
- Highest Productivity.
- Lowest Mining Cost.

A yearly production of 0.8 Mtpa was achieved with full production starting in Year 2022. A total of 10,814 kt of ore with an average Zinc grade of 31.85% and NSR10 value of $306/t was scheduled to be mined during the 14-Year mine life. During the mine life, a total of 2,087 kt of waste will be produced.

In the ore produced from designed stopes, a significant amount of economic grade material will be produced during stope and access development. This material is included as ore in the production schedule where the majority is defined as low-grade (NSR10≥$51/t and NSR10<$135/t). The planned Kipushi development and production schedules are summarised in Table 16.53 to Table 16.55.

Future proposed mine production have been scheduled to optimise the mine output and meet the plant capacity. The mining production forecasts are shown in Table 16.56. Mine, process and concentrate production are shown in Figure 16.84 to Figure 16.86.

Table 16.53 Kipushi Capital Development Schedule Summary

Capital	Total	Year −2	Year −1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13
Lateral Development																
Cap Lat Dev: Equivalent Metres (m)	12,365	1,446	3,250	1,922	1,705	282	80	157	1	87	351	668	771	1,015	436	192
5.3 m W x 5.8 m H	3,100	601	1,002	73	–	–	–	–	–	–	–	14	456	538	323	94
5.3 m W x 5.8 m H	5,970	512	1,420	1,209	1,362	213	80	56	1	67	280	415	205	147	–	–
5.3 m W x 5.8 m H	78	78	–	–	–	–	–	–	–	–	–	–	–	–	–	–
5.5 m W x 5 m H	22	22	–	–	–	–	–	–	–	–	–	–	–	–	–	–
6 m W x 6 m H	692	55	147	16	–	–	–	–	–	–	–	95	58	219	74	29
6 m W x 6 m H	983	43	223	290	143	40	–	78	–	20	61	23	23	–	–	40
6 m W x 6 m H	120	100	–	–	–	–	–	–	–	–	–	–	–	–	–	20
5 m W x 4.7 m H	410	–	120	149	33	–	–	–	–	–	10	–	7	65	16	10
5 m W x 4.7 m H	564	35	140	118	144	28	–	–	–	–	–	99	–	–	–	–
6.5 m W x 7.5 m H	244	–	152	–	–	–	–	–	–	–	–	–	23	46	23	–
6.5 m W x 7.5 m H	181	–	45	68	23	–	–	23	–	–	–	23	–	–	–	–
Vertical Development																
Cap Vert Dev: Equivalent Metres (m)	8,499	2,108	5,170	600	323	75	–	–	–	–	–	81	–	110	4	28
Pilot Holes	588	–	281	61	122	50	–	–	–	–	–	46	–	–	–	28
Raise Bore	4,317	954	2,856	367	140	–	–	–	–	–	–	–	–	–	–	–
0.9 m	2,551	853	1,698	–	–	–	–	–	–	–	–	–	–	–	–	–
1.5 m	412	–	170	103	61	26	–	–	–	–	–	35	–	14	4	–
2.5 m	397	300	–	–	–	–	–	–	–	–	–	–	–	97	–	–
3.5 m	165	–	165	–	–	–	–	–	–	–	–	–	–	–	–	–
5.0 m	69	–	–	69	–	–	–	–	–	–	–	–	–	–	–	–

Table 16.54 Kipushi Operating Development Schedule Summary

Operating	Total	Year –2	Year –1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14
Lateral Development																	
Op Lat Dev: Equivalent Metres (m)	20,396	50	1,362	2,501	1,996	1,454	1,099	1,022	1,165	1,408	1,430	1,695	1,620	1,360	1,349	700	184
5.5 m W x 5 m H	3,185	–	–	–	–	–	–	–	–	–	–	149	306	767	1,078	700	184
5.5 m W x 5 m H	1,355	40	239	370	110	–	48	42	60	155	70	164	59	–	–	–	–
5.5 m W x 5 m H	5,468	–	338	172	870	961	389	428	344	412	475	321	307	268	183	–	–
6 m W x 6 m H	505	10	367	88	–	–	–	–	20	20	–	–	–	–	–	–	–
6 m W x 6 m H	5,097	–	301	843	876	93	362	293	471	421	590	542	262	42	–	–	–
6 m W x 6 m H	2,365	–	–	909	80	400	60	71	–	330	64	155	102	106	88	–	–
8 m W x 6 m H	117	–	117	–	–	–	–	–	–	–	–	–	–	–	–	–	–
8 m W x 6 m H	2,264	–	–	120	60	–	240	188	270	70	231	364	584	137	–	–	–
8 m W x 6 m H	40	–	–	–	–	–	–	–	–	–	–	–	–	40	–	–	–
Vertical Development																	
Op Vert Dev: Equivalent Metres (m)	5,610	–	125	325	400	400	250	300	307	318	331	524	610	538	697	360	125
Raise Bore 0.9 m	5,610	–	125	325	400	400	250	300	307	318	331	524	610	538	697	360	125

Table 16.55 Kipushi Production Schedule Summary

Production	Total	Year –2	Year –1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14
Mined Material																	
Ore (kt)	10,815	5	101	799	799	799	799	799	790	799	799	799	789	799	800	802	335
Zn %	31.85	12.73	20.86	32.39	35.62	38.01	38.89	40.11	35.08	33.56	31.82	30.00	27.02	26.83	24.27	27.56	18.60
Zn Metal (kt)	3,445	1	21	259	285	304	311	321	277	268	254	240	213	214	194	221	62
Waste (kt)	2,087	141	425	295	241	80	38	65	41	68	96	151	155	145	81	57	8
Backfill																	
Fill Volume (000 m^3)	2,555	–	10	167	207	202	198	217	215	231	198	213	232	190	120	80	74
Fill Tonnes (kt)	4,952	–	20	333	409	399	388	420	416	442	381	411	441	360	231	157	142
Fill Tonnes (kt) – RF	1,178	–	–	–	–	–	–	96	86	163	160	54	219	222	102	25	52
Fill Tonnes (kt) – CRF	3,774	–	20	333	409	399	388	324	330	279	221	357	223	138	129	131	91

A series of figures follow showing the major schedule outputs. Note that the FS Early Works were scheduled to commence January 2021, which is effectively Year-1 of the schedule. Figure 16.75 to Figure 16.81 show the production tonnes and grade for the feasibility schedule.

Key points include:

- Production ramps up to the steady-state 800 kt/yr in Year-3.
- Schedule targets high grade first, with grade peaking in Year-7 as all high-grade areas are accessed and then drops off as lower grade areas are mined.
- Southern Zinc is mined after Big Zinc due to its lower grades.

Figure 16.75 FS Annual Ore Tonnes by Mining Area and Overall NSR10 Values

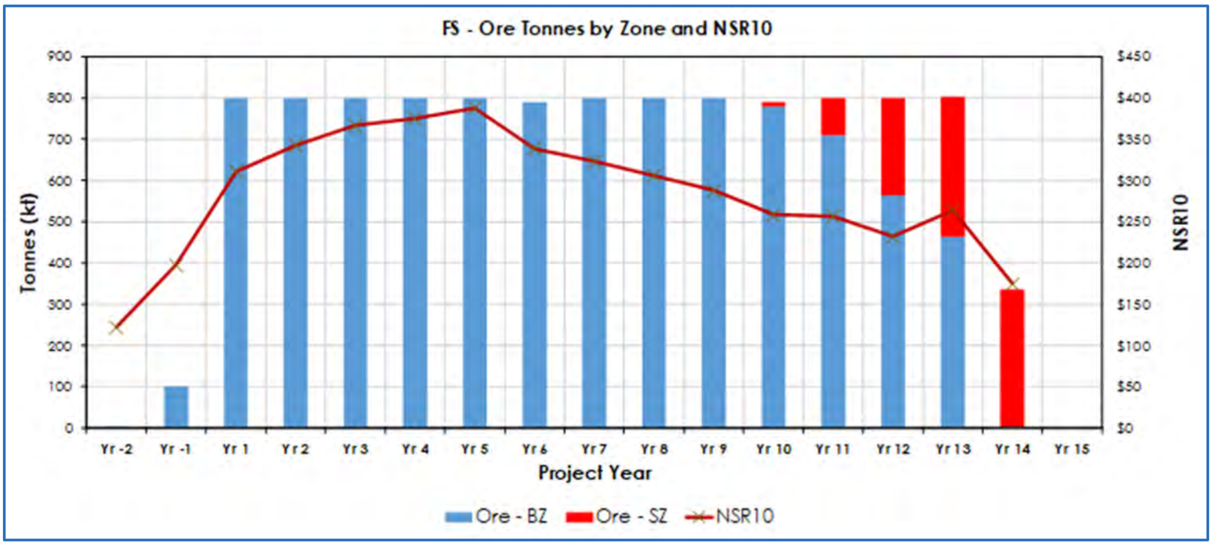


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Figure 16.76 FS Annual Ore Tonnes by Mining Area and Overall NSR13 Values

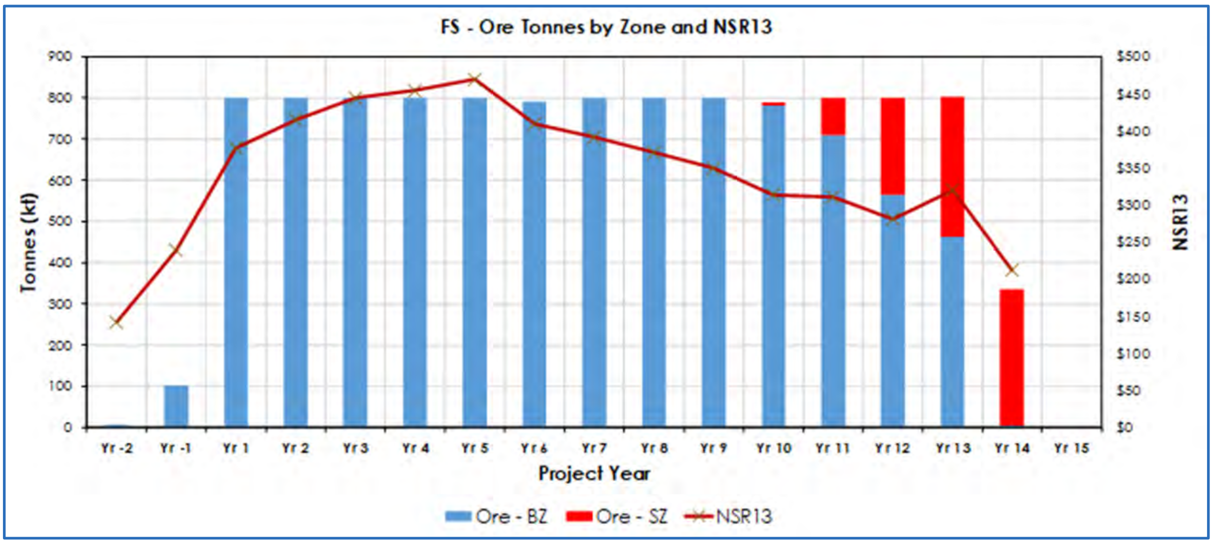

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Figure 16.77 FS Annual Ore Tonnes and NSR10 Values by Mining Area

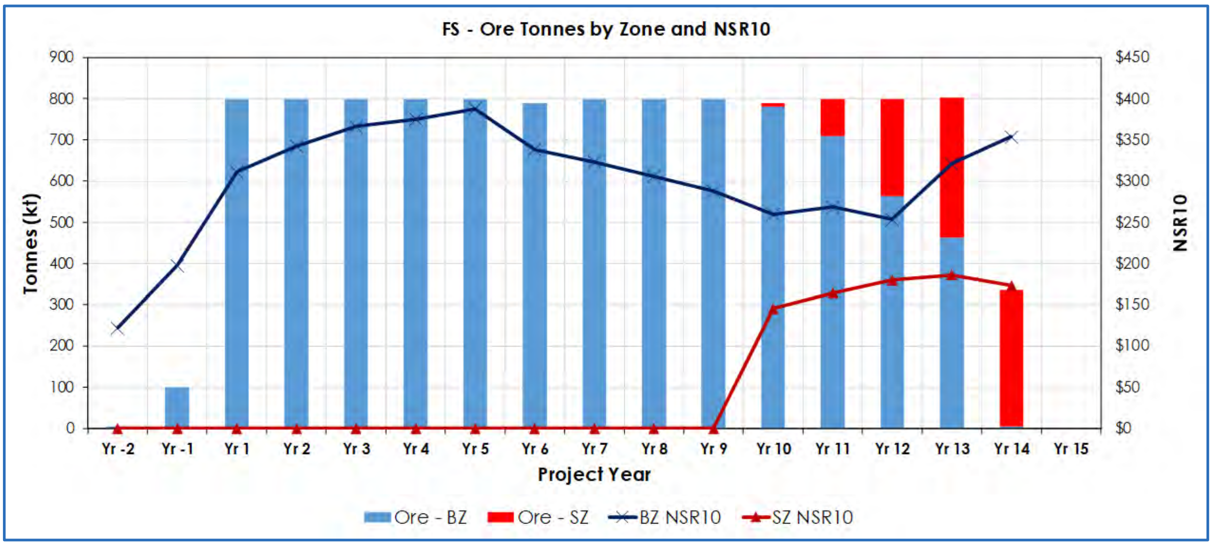

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Figure 16.78 FS Annual Ore Tonnes and NSR13 Values by Mining Area

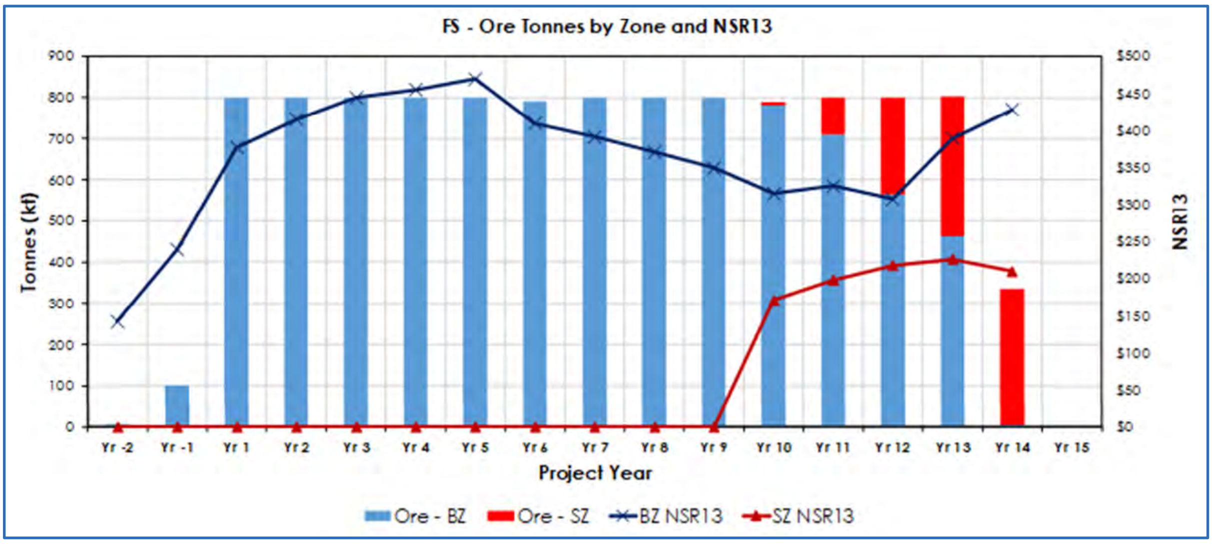

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Figure 16.79 FS Annual Ore Tonnes by Mining Area and Overall Zinc Grades

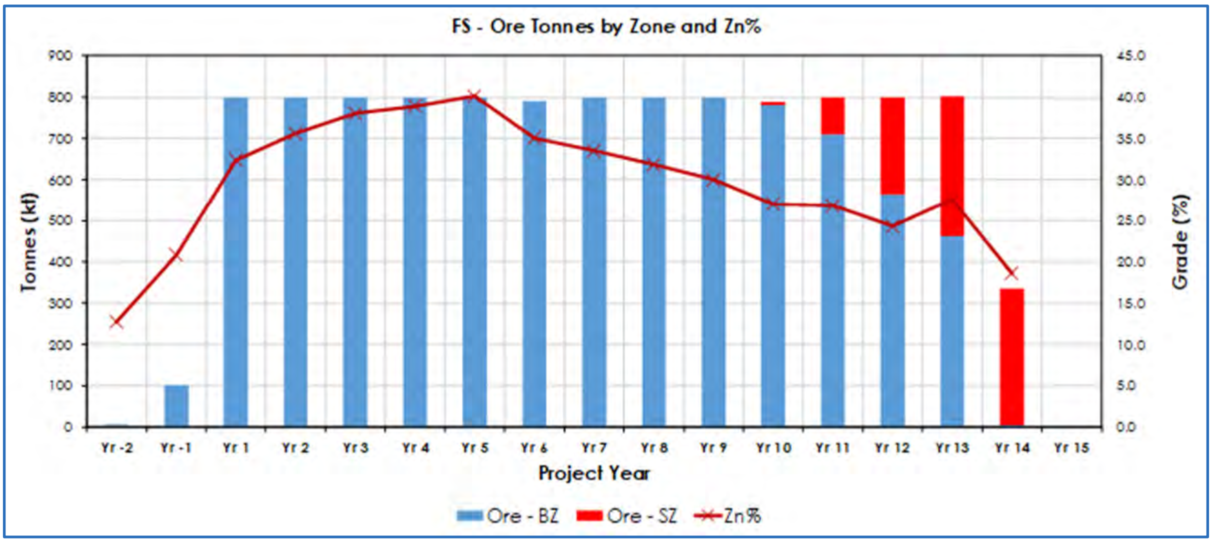

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Figure 16.80 FS Annual Ore Tonnes and Zinc Grades by Mining Area

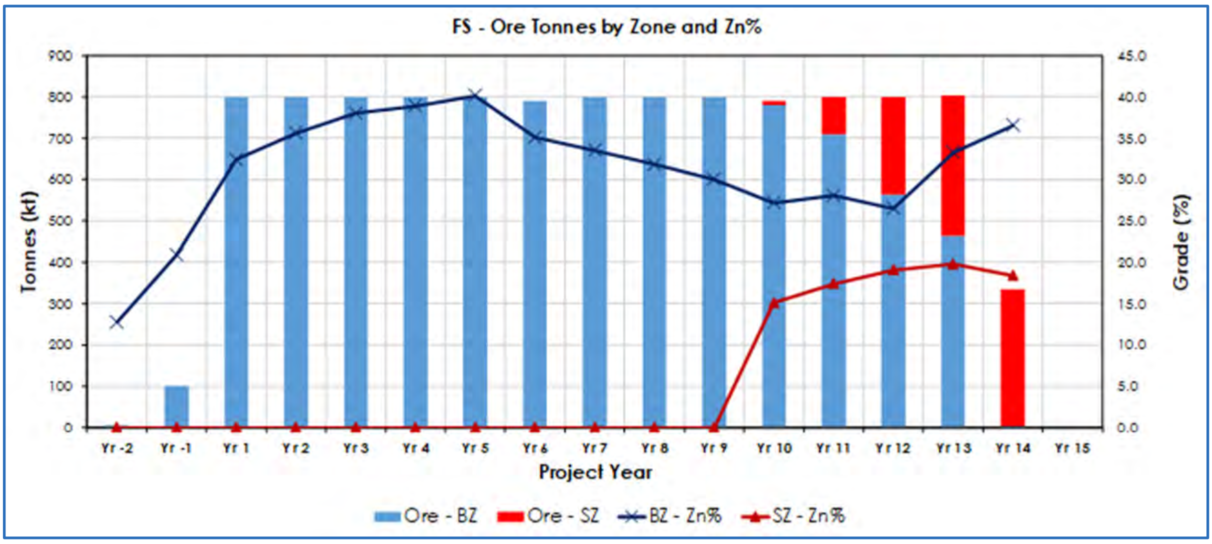

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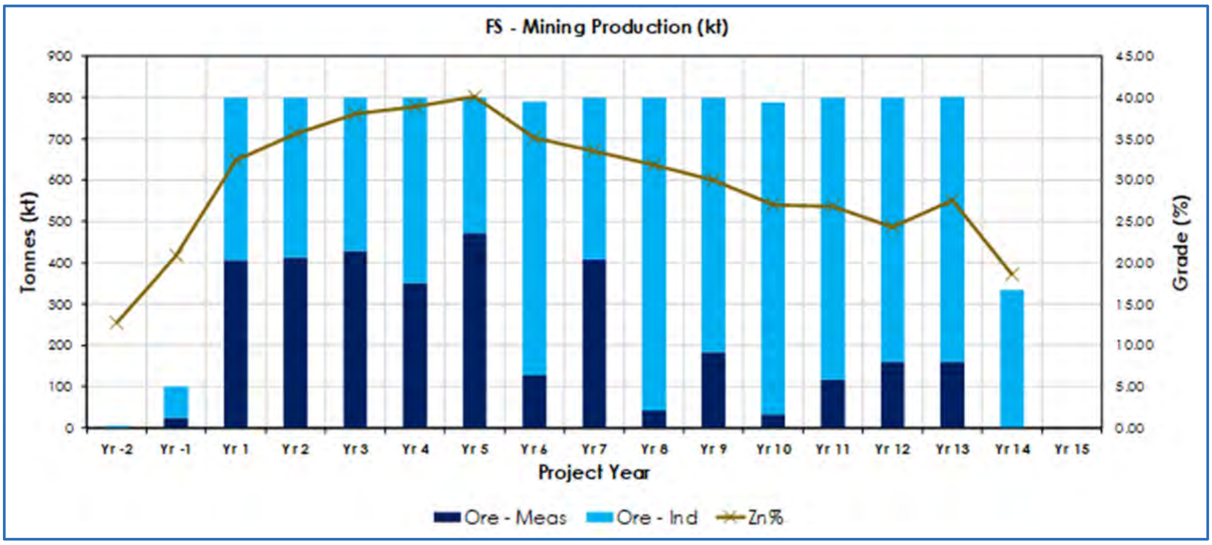
Figure 16.81 FS Annual Ore Tonnes by Resource Category



OreWin, 2022

Figure 16.82 and Figure 16.83 compare the production profiles of the FS against the PFS.

Figure 16.82 Comparison to PFS: Annual Ore Tonnes and NSR10 Values

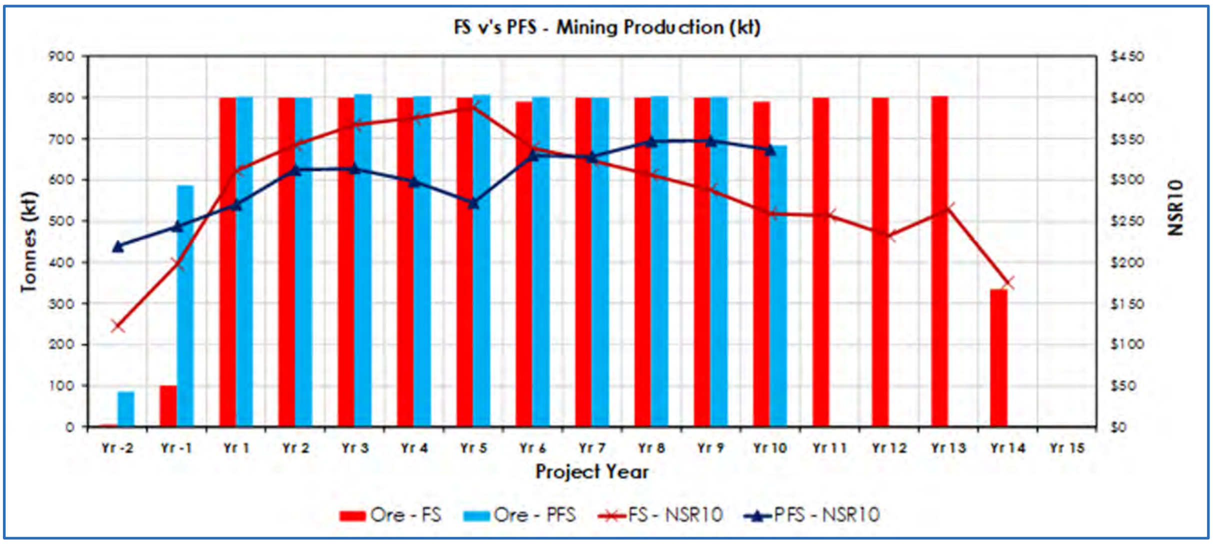


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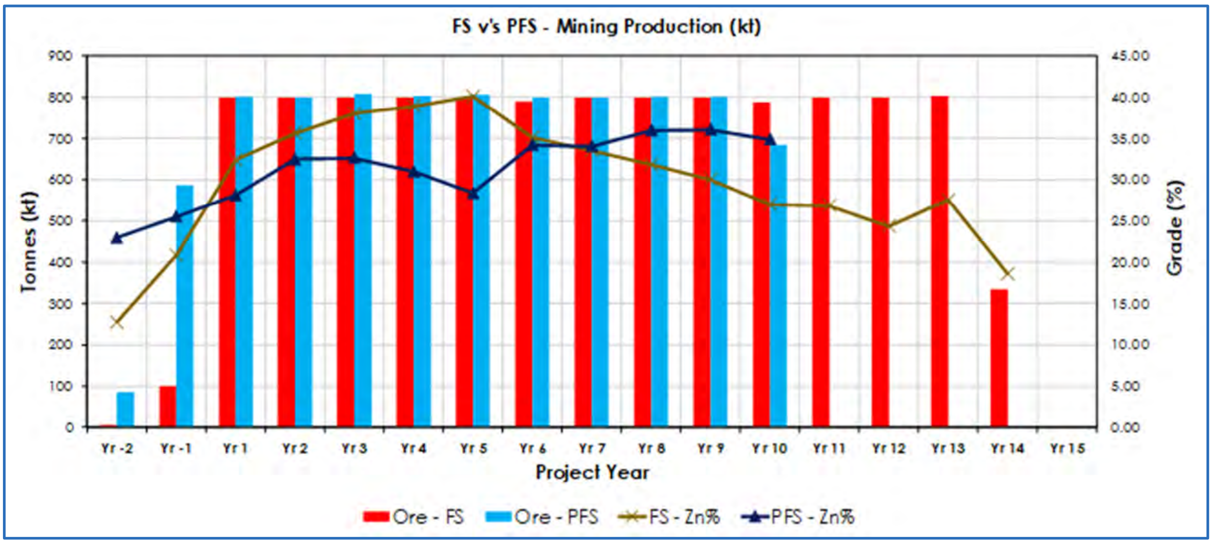
Figure 16.83 Comparison to PFS: Annual Ore Tonnes and Zinc Grades



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When compared to the PFS the Feasibility schedule has:

- A delayed production profile, due to constraints imposed by processing capacity.
- A longer production life due to the inclusion of Southern Zinc.
- A better grade profile, with a greater focus on mining high-grade areas first.

16.9.1 Production Summary

A summary of the production schedule over the first five years, and LOM is detailed in Table 16.56. Figure 16.84 outlines the LOM mined ore profile, while Figure 16.85 and Figure 16.86 show the LOM mill feed profile and LOM concentrate production respectively.

Table 16.56 Mining Production Statistics

Item	Unit	Total LOM	5-Year Annual Average	LOM Annual Average
Zinc Ore Processed				
Quantity Zinc Ore Treated	kt	10,814	792	787
Zinc Feed grade	%	31.85	36.43	31.85
Zinc Concentrate Recovery	%	95.63	95.87	95.63
Zinc Concentrate Produced	kt (dry)	6,013	508	437
Zinc Concentrate Grade	%	54.79	54.79	54.79
Metal Produced				
Zinc	kt	3,294	278	240

Figure 16.84 Mined Production

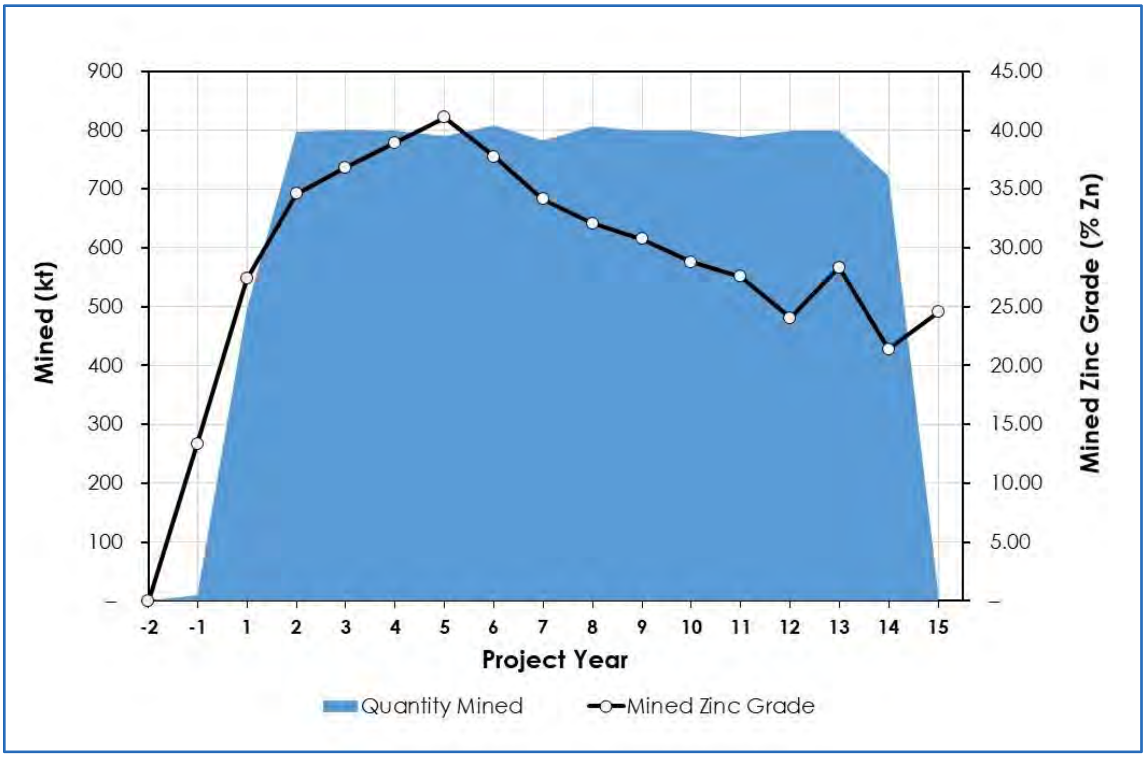

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Figure 16.85 Process Production

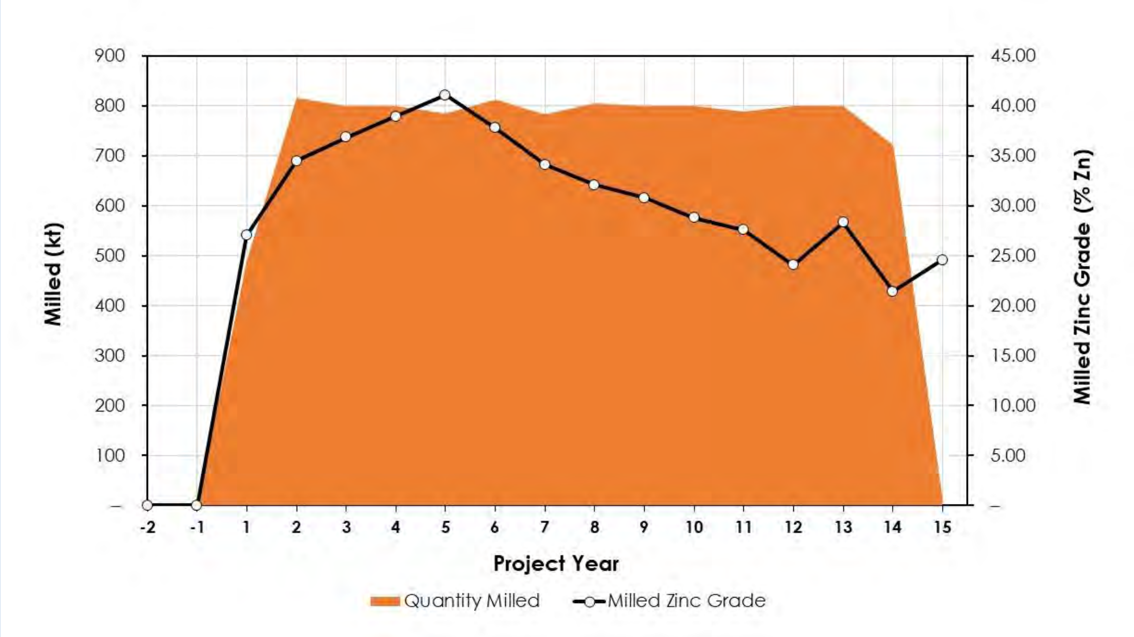

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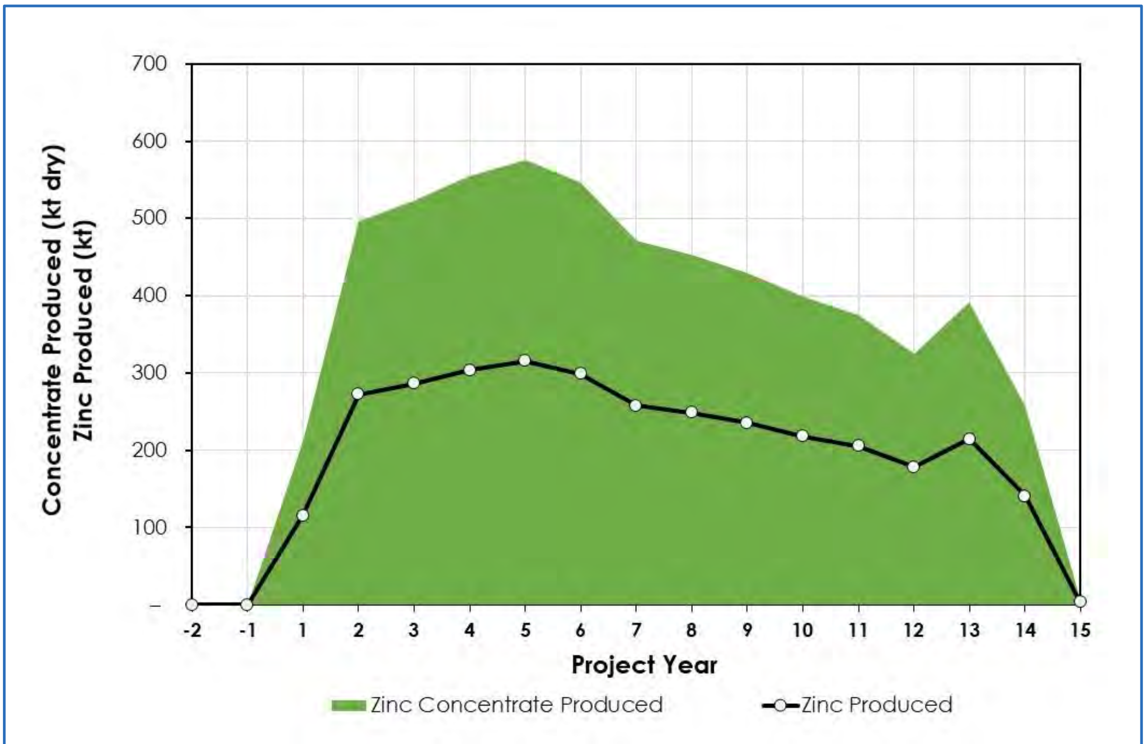
Figure 16.86 Concentrate and Metal Production



OreWin, 2022

16.10 Recommendations

This study demonstrates the feasibility of the project, so does not find any reason from a mining point of view that the project should not proceed.

16.10.1 Geotechnical

- Carry out stress measurements during the execution stage of the project.
- Areas where poor ground conditions occur should be identified and inspected regularly to confirm that the current support is appropriate and in reasonable condition. New support should be installed where required.
- As it is evident that poor ground conditions do exist on a local scale, pre-existing major excavations planned for use must be inspected and re-supported appropriately where required.
- Rock mass conditions should be verified by continuous mapping and inspections during construction.
- Ongoing geotechnical mapping should take place at regular intervals in the planned developments to verify the rock mass conditions determined and to assess the rock mass quality where there is currently little or no information. This will also allow for the identification of localised weak zones and potentially unstable wedges which should be appropriately supported.
- While the structural analysis provides an impression of the major joint sets across the project area, further geotechnical scanline mapping should be conducted regularly as mining commences to allow for the identification of low angle joints in the hanging wall, localised joint sets and for potential wedges or instabilities.
- Where good ground conditions exist, support recommended based on the results of the elastic modelling should be applied. Where poor ground conditions exist, it is recommended that a higher support class is used.

16.10.2 Mine Ventilation and Cooling

- The capacity and sustainability of pumping water from 850 mL should be assessed.
- Should the concern over pumping from 850 mL be realised, engineering optimisation for the re-location of the refrigeration plant to 1,150 mL should be completed.

16.10.3 Hydrogeology

- It is recommended that additional boreholes should be drilled. The purpose of the boreholes is:
  - To verify the simulated water level drawdown north of the Petite Conglomerate.
  - To check the influence of mining on the well field.
  - To check the water quality downstream of the TSF in comparison with upstream water quality.

- The model is a water management tool that should be updated regularly (every two years) to incorporate ongoing monitoring data and to adjust the model as additional information becomes available.
- The model should be updated sooner if any additional inflows or other changes are observed during mining. Immediate action should be taken as soon as changes (i.e. should high permeable zones be encountered during mining) are observed.
- It is recommended that an unsaturated flow model should be constructed to investigate in more detail and to include the effect of evaporation. For this simulation, evaporation was included in the sense that a lower flux into the tailings was specified. An unsaturated flow model will give a specified flux value with higher certainty.

16.10.4 Mining

- Preparatory work for the early works should be completed as soon as practicable.
- Alternative options for the backfilling strategy (described in this report) are traded-off against one another.
- Potential sources of waste to meet the shortfall investigated.

17 RECOVERY METHODS

17.1 Overview

The recovery methods described in this section relate to the new Kipushi Corporation SA's (KICO) Concentrator plant. The concentrator plant is designed to process a nominal 800 ktpa of run-of-mine (ROM) ore, from a high-grade zinc zone orebody of the KICO underground Zinc-Copper Mine in the Central African Copperbelt in the DRC, to produce a saleable zinc flotation concentrate.

Several metallurgical tests, as presented in Section 13 of this report, have been conducted on representative samples from the zinc orebody and formed the basis of the process plant design. The metallurgical test results indicate that the Kipushi mineralisation is amenable to a process involving pre-concentration by dense media separation (DMS) and bulk flotation to produce suitable saleable zinc sulfide concentrate.

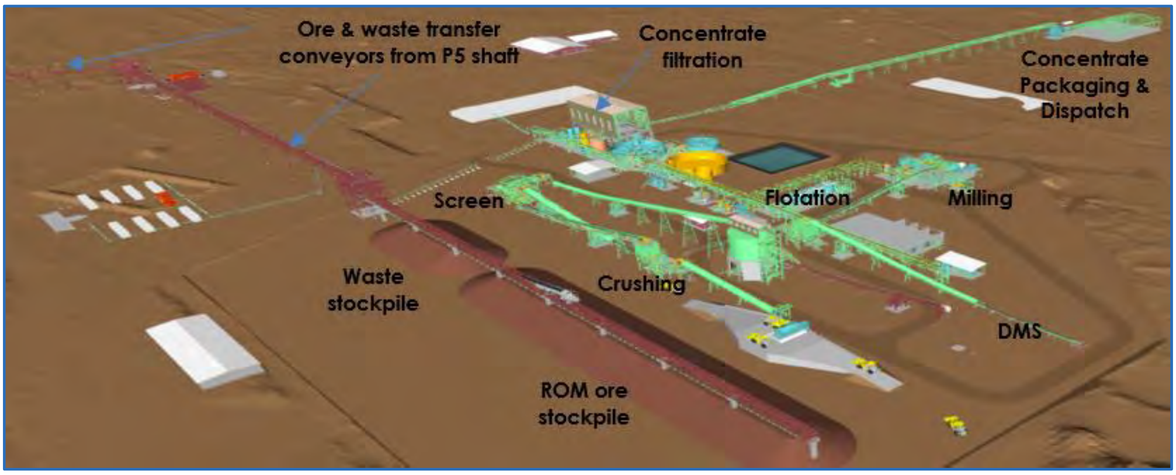
ROM ore and waste material from underground mining facility are hoisted through the existing P5 shaft system and transferred to the stockpiles area. Ore is withdrawn from the stockpile to feed the process plant, whilst reclaimed waste material is loaded into trucks for disposal.

17.2 Process Plant Site Layout Consideration

The new process plant, consisting of crushing and screening, DMS, milling, flotation, tailings and concentrate handling circuits, is located at the existing Kipushi Mine facility with limited free space within the brown field site.

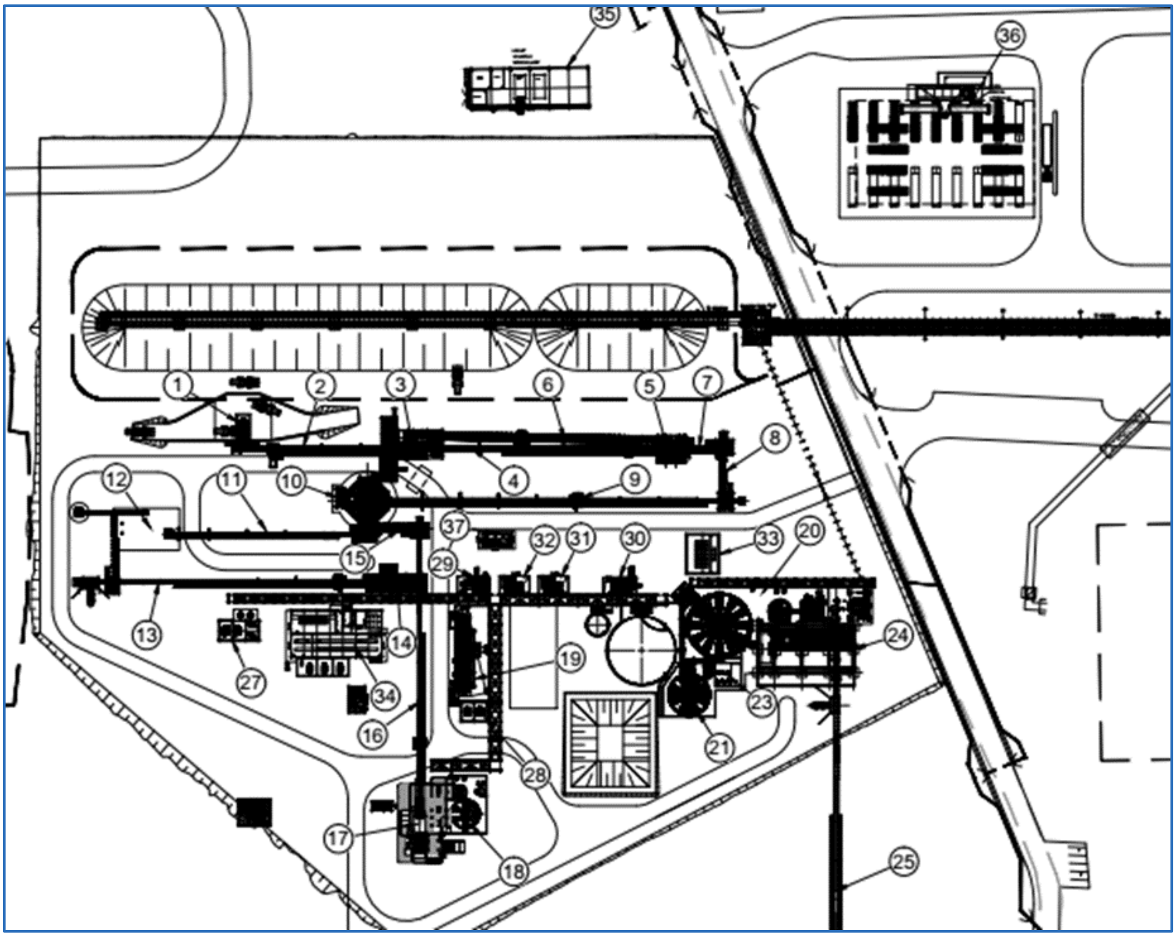
The development of the site layout design has been based on maximizing ease of operation and minimizing both the capital and operating costs. Key considerations in determining the optimum location for the process plant include safe transfer of ROM ore from the existing P5 shaft facility to the stockpile areas, minimizing demolition cost for existing structures, minimizing the need to pump, and optimizing the product handling and dispatch operations. The process plant 3D and layout designs are presented in Figure 17.1 and Figure 17.2 below.

Figure 17.1 KICO Process Plant 3D Layout



METC, 2022

Figure 17.2 KICO Process Plant Engineering Layout

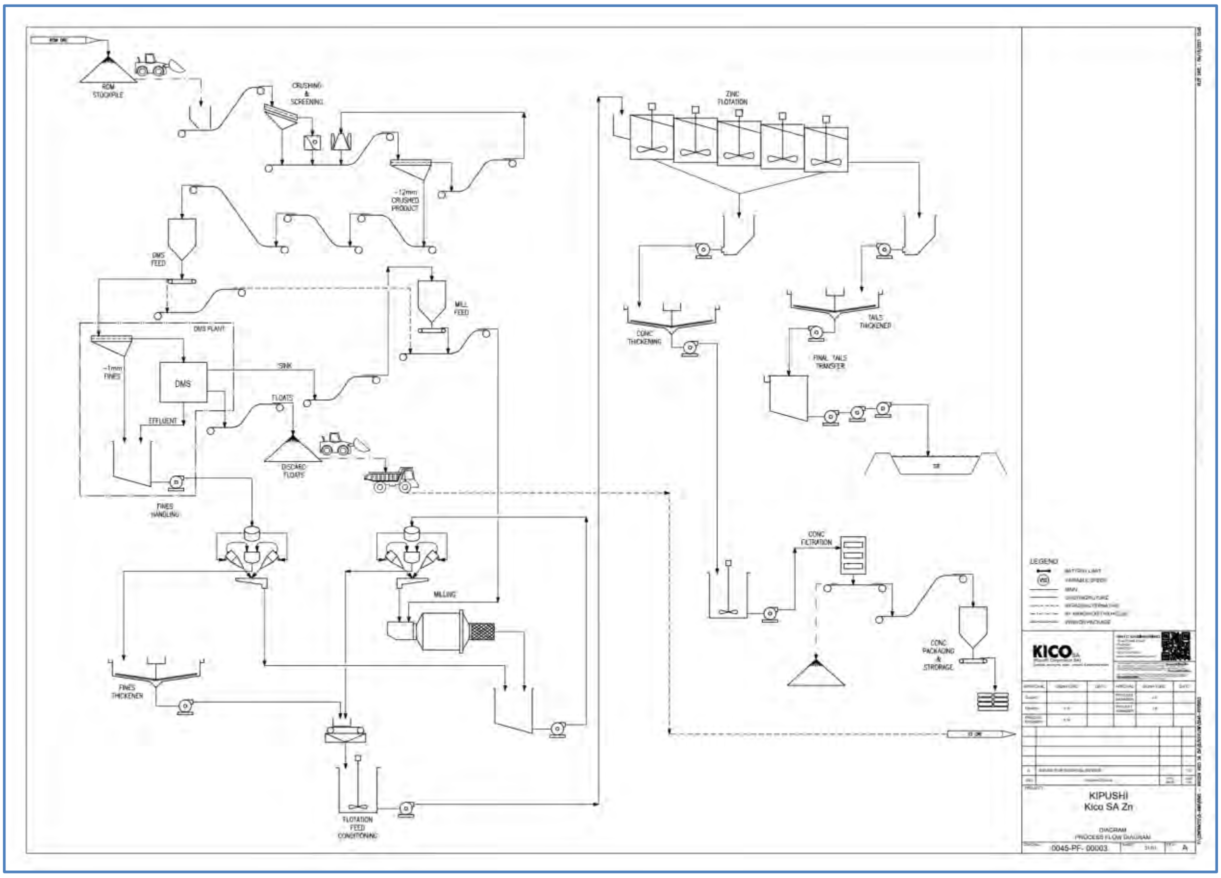


METC, 2022

17.3 Overall Process Flow Sheet

The proposed overall process flow circuit includes ROM crushing and screening, pre-concentration by DMS, ball mill grinding circuit, sulfide flotation, final tailings and concentrate handling facilities, air utilities, reagent and water services. The crushing plant product particle top size is set at 12 mm, whilst the milling circuit final grind size is at 80% passing 106 µm. The proposed process flow diagram is presented in Figure 17.3 below.

Figure 17.3 Process Plant Flow Diagram for the KICO Project



METC, 2022

17.4 Process Plant Design

17.4.1 Design Basis

The process plant design basis is summarised in Table 17.1 below.

Table 17.1 Process Plant Design Basis Summary

Plant Capacity	Units	Value
Annual throughput (dry solids)	t/a	800,000.0
Monthly throughput (dry solids)	t/m	66,666.7
Overall Plant Utilisation Factors		
Crushing Circuit	%	65.0
DMS Plant	%	86.5
Milling & Flotation Circuit	%	86.5
Plant Nominal Solids Throughput Capacity		
Crusher plant	t/h	140.5
DMS Plant	t/h	105.6
Milling and Flotation Circuit	t/h	82.8
Design Feed Grade – (Based on Mine LOM average plan)		
Zn	%	31.9
Cu	%	0.7
Sulfides	%	65.4
Overall Metallurgical Performance		
Zn HLS Recovery (stage recovery)	%	98.9
Zn Flotation Recovery (stage recovery)	%	96.4
HLS Concentrate Mass Pull Yield	% (m/m)	73.1
Flotation Concentrate Mass Pull	% (m/m)	74.3
Zn Final Concentrate Grade	%	54.8
Process Plant Circuits – Key Design Basis		
Bond Crusher Work Index (Cwi)	kWh/t	12.9
Crusher Plant Feed Particle size	mm	280.0
Crusher Plant Product particle size, P_{100}	mm	12.0
DMS Feed Screen Undersize (–1 mm fines)	% m/m	26.0
DMS cut density	t/m3	3.1
Bond Ball Work Index (BBWi)	kWh/t	8.2
Mill Final Grind Product Size, P_{80}	µm	106.0
Rougher Flotation Residence Time	minutes	25.0
Cake moisture	% m/m	9.0
Bulk bag size	t	2
Final Concentrate Nominal Rate	t/h	58.8

17.4.2 Design Assumptions and Key Parameter

Testwork data has been used for the design principles, however some design assumptions have been made as follows:

- ROM ore feed composition has limited, if at all, clay material.
- For the purpose of sizing the ball mill, it was assumed that the DMS concentrate would exhibit similar milling characteristics to the ROM ore.
- The thickener design is based on solids flux rate at 6 t/(m^2.day) and a rise rate 0.73 m/h.
- A concentrate filtration flux rate of 406 kg/(m^2.h) and a final filter cake moisture content of 9% formed basis for the filter design.
- The concentrate produced is free flowing, non-reactive and does not age harden.
- Mass flow is achieved in all bins.

Key parameters used in the development of the plant design and operating costs are summarised in Table 17.2.

Table 17.2 Key Design Basis Parameters

Grinding And DMS Consumables	Units	Value
Steel Balls Consumption rate	kg/t	0.74
DMS Media – Ferro Silicon Consumption rate (based on DMS feed tonnage)	g/t	300.0
Reagents And Consumables		
Collector SIPX Consumption rate	g/t	40.0
Frother MIBC Consumption rate	g/t	36.0
Activator – Copper sulfate ($CuSO_4$) Consumption rate	g/t	900.0
Hydrated lime – $Ca(OH)_2$ Consumption rate	g/t	800.0
Flocculant Consumption rate	g/t	13.3

17.5 Process Description

17.5.1 Ore Receiving

Ore and waste are crushed underground to a product top particle size at P_{100} of 280 mm and hoisted to surface using the refurbished P5 Shaft system.

Both crushed ore and development waste will be intermittently (and separately) hoisted to surface, depositing into a single bin on surface, within the P5 Shaft headframe. Material is reclaimed from the bin via a vibrating feeder, which discharge onto a single existing 900 m overland conveyor T5A connecting Shaft 5, via a series of four overland transfer conveyors (T5, T6, T7 and T8), to the waste and ore stockpiles area at the Old Kipushi Concentrator (OKC).

The transfer overland conveyor T8 is designed and equipped with a movable tripper car system that allows discrete discharging of waste and ore onto separate waste and ore stockpiles, respectively. The stockpile area is designed to allow reclaim of material by use of Front-end loaders (FEL). Level transmitters and position switches will be installed along the travel length of the T8 tripper conveyor system for stockpile level and tripper position monitoring.

17.5.2 Crushing Plant

The crushing and screening circuit receives ROM ore onto the ore stockpile, via a series of overland transfer conveyors, as primary crusher product, from the Kipushi underground mine.

Ore is withdrawn from the ore stockpile using a FEL to feed a two-stage crushing plant at a nominal solids feed rate of 140 t/h. The crusher circuit design has been set up to minimise the production of fines (–1 mm). To this end, an open circuit secondary jaw crusher is used in conjunction with a closed-circuit tertiary cone crusher.

Screened crushed ore product, at –12 mm size fraction, is transferred and discharged into a DMS feed bin from where the DMS plant is fed.

17.5.3 DMS Plant

Crushed ore from the DMS feed bin is withdrawn and screened to produce a –12 + 1 mm size fraction product to feed the DMS. Screen undersize as –1 mm fines, together with the DMS effluent slurry, is transferred to the fines handling circuit located at the milling area.

The DMS plant uses atomised ferrosilicon as 'medium', plant cut-point density of 3 t/m^3. The average DMS feed grade is 31.9% Zn and is upgraded to about 47% Zn in concentrate at an average mass pull of approximately 71% to sinks product.

DMS cyclones are used to concentrate the zinc ore, with concentrate reporting to the cyclone underflow (referred to as sinks), and the lighter minerals passing through the cyclone overflow (referred to as floats). Each cyclone stream is subsequently screened to ensure the media (FeSi) is washed and recovered from the ore streams. The washed DMS product is transferred via a conveyor to the mill feed bin. The DMS floats stream is conveyed to the DMS discard stockpile.

The DMS media (FeSi) density is controlled with densifiers and magnetic separation drums. FeSi media make-up is done manually.

17.5.4 Fines Handling

The fines handling circuit receives the DMS effluent stream, together with –1 mm fines fraction from the DMS preparation screen. The fines handling circuit feed stream is pumped to a dewatering cyclone cluster, with the cyclone overflow gravitating to the fines thickener feed box whilst the cyclone underflow gravitates to the mill discharge sump.

The fines thickener feed is thickened to 55% solids by mass and transferred to the flotation conditioning tank via a trash linear screen. Design allows for a flexibility to pump thickener underflow to the mill discharge sump.

17.5.5 Milling Circuit

DMS concentrate from the sinks screen oversize is transferred to the mill feed bin. The mill is fed from the bin at a controlled rate, with steel balls added manually onto the mill feed conveyor. DMS effluent, together with –1 mm fines, is pumped into the mill discharge sump.

The milling circuit is designed as a closed-circuit variable speed ball mill with a classification cyclone cluster. The milling circuit comprises a single 1,1 MW ball mill. The milling circuit is designed to achieve a P_{80} of 106 µm. The cyclone overflow gravitates to the flotation feed tank via a trash linear screen.

17.5.6 Flotation

Mill cyclone overflow of P_{80} passing 106 µm, together with the fines thickener underflow, is transferred to a linear trash screen prior to collecting into a zinc flotation feed conditioning tank. The flotation feed slurry is subjected to the flotation process where the targeted zinc sulfide minerals are concentrated and separated from the gangue material.

A single stage bulk sulfide rougher flotation is employed to upgrade the Zn. The flotation objective is to recover zinc in the froth product and reject both pyrite and dolomite to tailings. Most of the copper and lead in the feed will float with the zinc concentrate.

The flotation circuit comprises a single bank of 5 off 20 m^3 identical flotation cells sized to achieve a design residence time of 25 minutes, dedicated concentrate and tails pump transfer systems, as well as the blower air supply system and the reagent dosing systems.

The flotation circuit is operated at pH 11.5 to depress pyrite from floating with the concentrate. The reagent regime includes a xanthate collector (SIPX), copper sulfate to activate the sphalerite and a frother to assist with stable froth formation.

17.5.7 Concentrate Handling

The Zn flotation concentrate is thickened and filtered ahead of bagging in a semi-automated bagging facility. Concentrate thickener underflow is filtered using a vertical tower filter press to produce a concentrate filter cake with 9% moisture content. Concentrate thickener overflow water is transferred to the process water circuit.

The concentrate bags are stored in dedicated storage bays with facility to load onto trucks. Each bag is sampled and tagged ahead of dispatch.

17.5.8 Waste Handling

Waste handling includes development waste from underground mining operations, DMS floats discard, as well as flotation tails, which are thickened and pumped to the tailings storage facility (TSF). Tailings thickener underflow is sampled for metallurgical accounting purposes, at the point where it discharges into the final tailings transfer tank.

Coarse waste (mine development waste and DMS residue) is to be used for mine stope backfilling. The waste material is stockpiled and transferred from the temporary stockpile areas to long term stockpiles, utilising mobile equipment by a third-party contractor.

17.5.9 Utilities

17.5.9.1 Water

Raw and Potable Water Services

There are two main sources of water used in the process plant area, namely;

- Mine underground dewatering water, pumped through P5 shaft mining area.
- Potable water from existing water well field.

Underground water will be pumped from P5 via a 350 NB line to a raw water tank located at the new water management area, from where it will be distributed to the various users, with the balance discharged to the storm water drains.

The underground water quality is not suited for use in concrete making for the CRF facility, fire water or gland service water. Metallurgical testwork also indicated that underground water negatively impacted metallurgical performance, and therefore could not be utilised as process water make up. For this reason, underground water from the new raw tank is used mainly for flushing and hosing at the CRF facility and at the process plant, as well as dust suppression water on transfer conveyor discharge points, stockpiling area and on mine roads by use of water tankers.

The potable water quality from the existing borehole well field is considered adequate for process top-up water, gland service water and reagent make-up water requirements in the process plant. Potable water is received from the water supply interface point and delivered to a potable water tank and a fire water tank located at the water management area.

From the potable water tank, water is distributed to the process plant area to service all the safety showers around the plant, to top-up the process water tank and the plant potable water tank. A separate, dedicated set of pumps is provided in the design to supply potable water to admin and general infrastructure building around the P5 shaft area.

Process Water

The process water circuit receive water from all the thickener overflow streams in the plant, as well as top-up water from the potable water supply line from the borehole well field. A raw water top-up line is also provided as an alternative source in case of potable water supply interruptions. Process water is used mainly for dilution and as spray water in the milling and flotation area.

Gland Service Water

The gland service water system uses a potable water quality, and it comprises a low-pressure gland service system for most of the slurry pumps and a high-pressure gland service supply system for the tailings pump trains.

Fire Water

Potable water is pumped from the potable water distribution tank to the fire water storage tank. From this tank water is distributed for fire suppression purposes to the P5 rock handling area, P5 shaft area, main mine area as well as the concentrator complex.

A jockey pump, electrical pump and back-up diesel pump is included in the design for the fire water distribution purposes. Provision has also been made for the fire water tank level top up by mobile water tanker, as well as top up by raw water if the potable water plant is not accessible.

17.5.9.2 Air Services

Compressed instrument and plant air systems are supplied from a common compressor set consisting of a duty and standby units rated at eight bar pressure. Dedicated air receivers are provided for the supply and distribution through the plant of plant air and instrument air, respectively. Provision is made in the design for stand-alone, dedicated compressors to supply air for pressing and drying applications in the filtration process, respectively.

A dedicated blower set comprising of duty and standby units is provided to supply low-pressure blower air to the flotation circuit.

17.5.9.3 Reagents

The Kipushi process plant utilises several reagents to ensure that the required zinc recovery and metallurgical efficiencies are achieved. The reagent types include collector, activator, frother, lime and flocculant.

All reagent make-up facilities, except for the frother pumping facility, comprise a day tank, with at least 24-hour storage capacity, and a mixing tank.

Copper Sulfate – Activator

Copper Sulfate ($CuSO_4$) is delivered to site in 1 t pallets containing 25 kg bags. The bags are manually transferred from the storage area to the mixing plant by means of a forklift. The copper sulfate is mixed with potable water to produce a reagent solution at 20% w/w reagent strength. When the mixing process is completed, the solution is transferred to the day storage dosing tank for distribution.

Collector – Sodium Isopropyl Xanthate (SIPX)

Sodium Isopropyl Xanthate (SIPX) is utilised as a sulfide collector, targeting Zn sulfide (sphalerite). SIPX is supplied in powdered form in 25 kg bags. The SIPX is kept within a dedicated area of the reagent store and is transported via forklift to the mixing facility. The SIPX is mixed with potable water to produce a reagent solution at 5% w/w reagent strength. Once the solution is appropriately mixed it is transferred to the day storage dosing tank for distribution.

The SIPX storage area and process design cater for the flammable nature of this material.

Frother (MIBC)

Frother, methyl isobutyl carbinol (MIBC), is received in liquid form at 100% pure strength and does not require further dilution. The frother is received in 1m^3 ISO containers and is pumped directly from the container to the usage point(s) in the flotation circuit.

Lime

Hydrated lime is utilised as a pH modifier in the flotation circuit. The lime is delivered to site in 1 t bags, which are stored in a dedicated area within the reagent storage facility. The bags are transported by forklift to the mixing facilities. The lime is mixed with process water to a 20% w/w reagent strength slurry prior to transfer into a day storage dosing tank for distribution.

Flocculant

Flocculant is utilised at the fines thickener, concentrate and tailings thickeners. The flocculant is delivered in 25 kg bags and is made up to a solution strength of 0.5% w/w at make-up stage and is further diluted inline to 0.05% w/w prior to dosing into thickeners. After the required hydrolysis time, the activated flocculant at 0.5% w/w is pumped to the flocculant storage tank for distribution to the three thickeners.

17.6 Equipment List

The plant consists of the following major mechanical equipment as listed in Table 17.3 below.

Table 17.3 Major Equipment List

Number	TAG No	Equipment Name	Equipment Sizing / Specification	Motor kW
1	3220-CRC-001	Secondary Crusher	Jaw Crusher, 80–120 t/h, 50 mm CSS	110
2	3220-CRA-001	Tertiary Crusher	Cone Crusher, 140–170 t/h, 12 mm CSS	220
3	3210-SCV-001	Crushed Ore Classification Screen	281 t/h, 40 mm Top deck, 12 mm Bottom deck, 1.2 m W x 3.66 m L	11.3
4	3320-PKE-001	DMS Plant Package	106 t/h feed, mass pull at 71%, (include prep. Screen)	450
5	3420-MLB-001	Ball Mill	3.6 m Diameter x 5.5 m L, Overflow discharge	1100
6	3430-THB-001	Fines Thickener	11 t/h, 55% m/m u/flow, 9 m Diameter, MOC – Mild Steel	4.0
8	3532-FTA-001-005	Zn Rougher Flotation Cells	5 off 30 m³ Tank Cells, 100 t/h, 216 m³/h @ 1.28 SG, 25 min Residence time	45/cell
9	3611-THB-001	Zn Concentrate Thickener	59 t/h, 55% m/m u/flow, 20 m Diameter, MOC – Mild Steel	7.5
10	3621-PKE-001	Zn Concentrate Filtration Package	Vertical Tower Pressure Filter, Cake moisture ~ 9% w/w	131
11	3641-PKE-001	Final Zn Concentrate Packaging Plant	Bagging plant, nominal 59 t/h conc, based on 1.5 t bulk bag size, auto bagging system	44
12	3711-THB-001	Tailings Thickener	24 t/h, 55% m/m u/flow, 13 m Diameter, MOC – Mild Steel	5.5

17.7 Plant Ramp-up

Mining activities will begin before the process plant is constructed or ready to receive ore and thus, strategic ore and waste development stockpiles have been allowed for in the design of the surface infrastructure. The plant's ramp up / strategic stockpile size is constrained by available space.

- A plant-ramp up profile for ROM throughput and the concentrate produced, has been developed using a typical McNulty type ramp-up curve for a relatively simple plant. The proposed ramp-up curve is illustrated Figure 17.4.

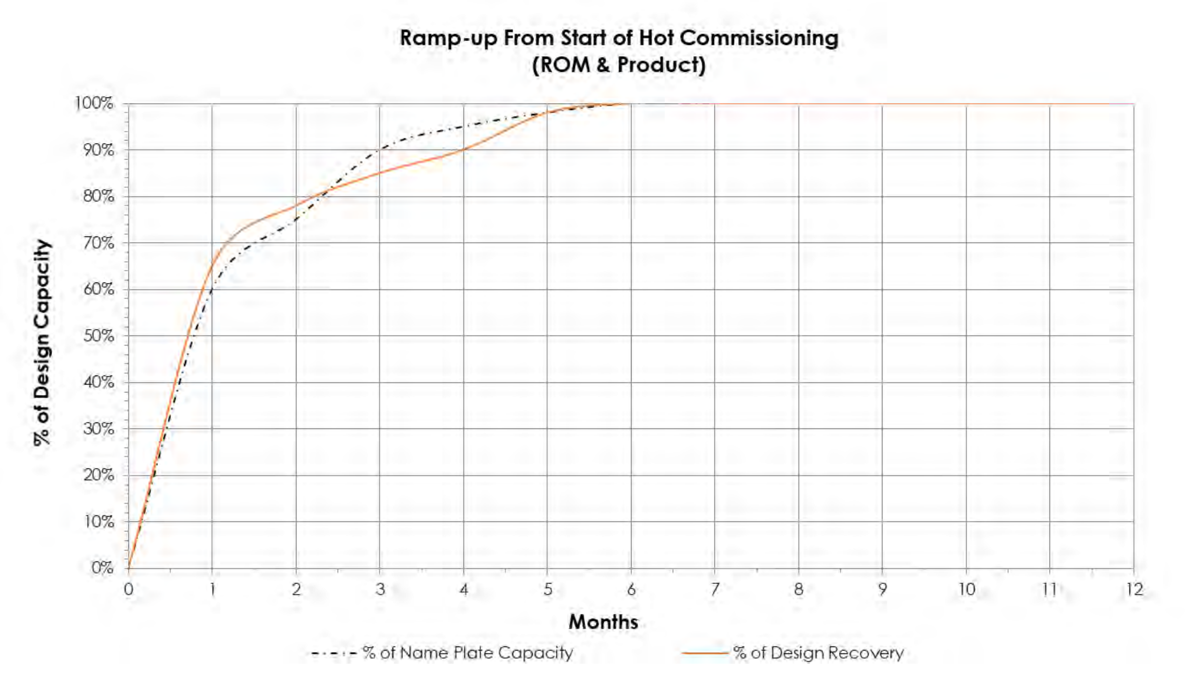
Once the plant is constructed and cold commissioning is completed, it is estimated that:

- After two and a half months, the plant should be able to meet 80% of its ROM nameplate capacity consistently. Full throughput should be consistently achieved after six months of full operation, assuming mining has progressed appropriately.
- Design zinc recoveries should be achieved consistently in month six. Zinc recovery may be impacted by:
  - Water quality and reagent consumption and control strategies.

- The grind size and ore mineralogy.
- Zinc feed grade variability.

- The plant design is based on an overall plant availability of 86.5% for the milling and flotation circuits, and thus, there is some capacity for catch-up built into the design. Whilst a normal availability of around 90% could be expected for a mill/float circuit, it is relevant to note that there is limited buffering capacity between plant sections and the power supply reliability is lower than average.

Figure 17.4 ROM and Product Ramp-Up from Start of Hot Commissioning



METC, 2022

17.8 Comments on Section 17

Outcome of the extensive testwork completed for the Kipushi Project has been used to update the process flowsheet, estimate zinc recovery and improve understanding of the interrelation between DMS, Milling and Flotation. Changes to the flowsheet have included:

- Reducing the DMS Feed size to 12 mm from 20 mm so that the DMS concentrate top-size is also suitable ball mill feed without the need for further crushing. The reduced top-size increases the fines proportion of the crushed ore and this increases the mass fraction reporting to flotation feed. However, milling efficiency and grind consistency is improved significantly.
- Simplifying crushing and screening.
- Modifying the handling of fines and flotation feed in the circuit to optimise circuit flexibility.

- Modifying stockpiles and plant layout to improve operability.
- Removal of the Pb-Cu flotation stage as this results in excessive zinc losses. The previously lost zinc, together with relatively low levels of Cu and Pb, now report to final zinc concentrate providing a net revenue increase. An added benefit is that capital cost and reagents associated with the Pb-Cu float (such as cyanide and the Pb-Cu collector) are no longer required.

The improvements to the flowsheet in the FS have resulted in a simpler and more robust processing circuit and this has been enhanced by an improved plant layout.

18 PROJECT INFRASTRUCTURE

18.1 Project Infrastructure Summary

The Kipushi Project is located within the town of Kipushi in the south-western part of the Haut-Katanga Province in the Democratic Republic of Congo (DRC) and adjacent to the border with Zambia and shown in Figure 18.1. The Kipushi town is situated approximately 30 km south-west of Lubumbashi, the capital of Haut-Katanga Province. Kipushi is connected to Lubumbashi by a paved road. The closest public airport to the Kipushi Project is at Lubumbashi where there are daily domestic, regional, and international scheduled flights. As part of the Project, the 34 km rail spur connecting the Kipushi Station to Munama will be reinstated to facilitate transport of concentrate.

Shaft 5 and the surface infrastructure associated at the Old Kipushi Concentrator (OKC) (location of the proposed processing plant) reside within two separate and discrete fenced areas in the town of Kipushi. The two demarcated areas are linked by an existing pipe rack and an underground cable tunnel that crosses through 300 m of public space and over one public road.

The project infrastructure relates to the surface component of operational support systems covering all mine equipment and associated buildings, outside of what has already been defined as part of mining and processing directly responsible items.

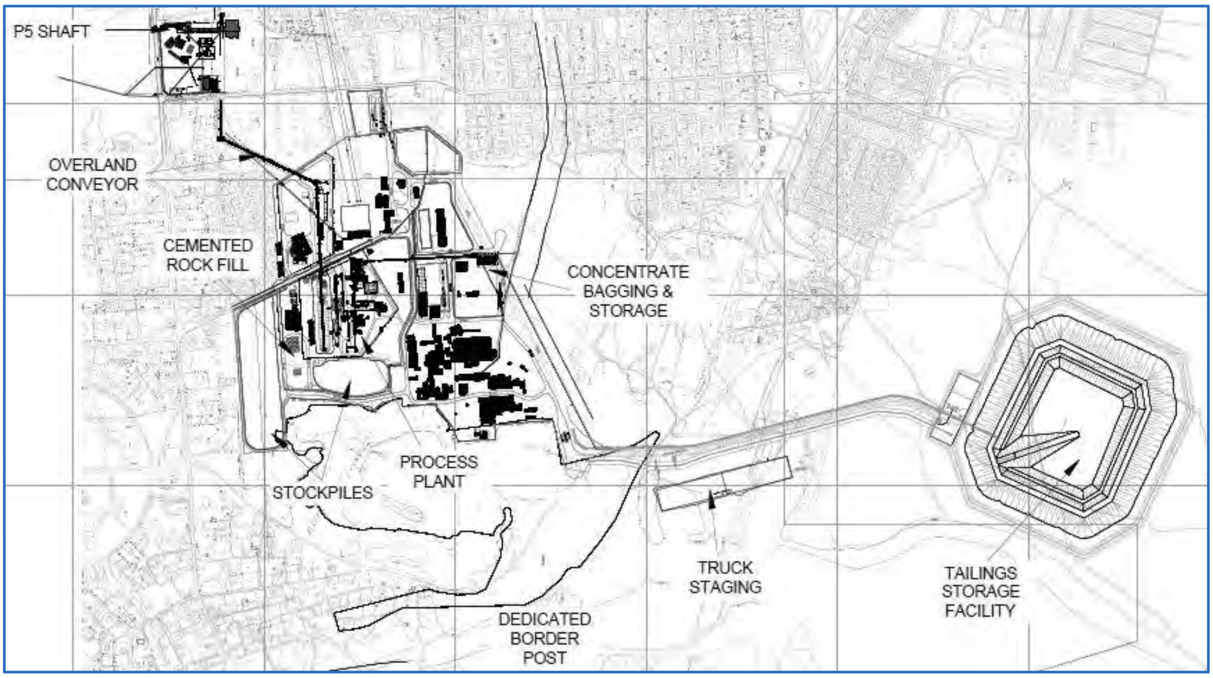
The large site, two distinct different working areas, its historic brownfield nature, and its tight enclosure within the town of Kipushi, make infrastructure more complicated than many other typical mining operations.

The property hosts surface mining and processing infrastructure, offices, workshops, stores, and connection to the national power grid. The property also hosts historical infrastructure, such as a mineral processing/beneficiation plant, that will not be further utilised. Most of the surface infrastructure is owned by Gécamines and was either transferred or is leased to KICO. Key aspects of the project infrastructure are:

- Electricity is supplied by the state power company of the DRC, Société Nationale d'Electricité (SNEL), using two transmission lines from Lubumbashi. There are pylons in place for a third line. The lines will be refurbished and re-stringed with aluminium conductors to minimise copper theft incidents.
- 14 MW of back-up power will be provided on site (new diesel gensets).
- The refurbishment of the diesel tank farm.
- Communications infrastructure required to support an operating mine.
- Leased and refurbished accommodation in Kipushi for owner's team personnel.
- A new overland conveyer system for transporting ore and waste from Shaft 5 to the new plant/ore stockpile and temporary waste storage area, respectively.
- A run-of-mine ore stockpile and a temporary waste stockpile area.

- A new processing plant and supporting surface infrastructure that incorporates the following unit operations:
  - Crushing and screening.
  - Dense media separation (DMS) to remove dolomitic wastes for backfill.
  - Milling.
  - Bulk flotation with pyrite rejection.
  - Concentrate bagging facility.
- A new tailings dam with an overhead line supplying power to the facility.
- A new backfill plant and supporting surface infrastructure that incorporates the following unit operations:
  - Crushing and screening
  - Cement silos/storage
  - Cement slurry plant
- A new border post and infrastructure to facilitate exports of zinc concentrate from the Kipushi border along the road to Solwezi in Zambia.
- Old (refurbished) and new facilities including:
  - General office, technical buildings, and structures.
  - Mine services buildings (change rooms, mess, kitchen, laundry).
  - Workshops, stores, and construction laydown areas.
  - General electrical buildings.
  - Security and emergency services buildings.

Figure 18.1 Overall Proposed Site Layout



Ivanhoe, 2022

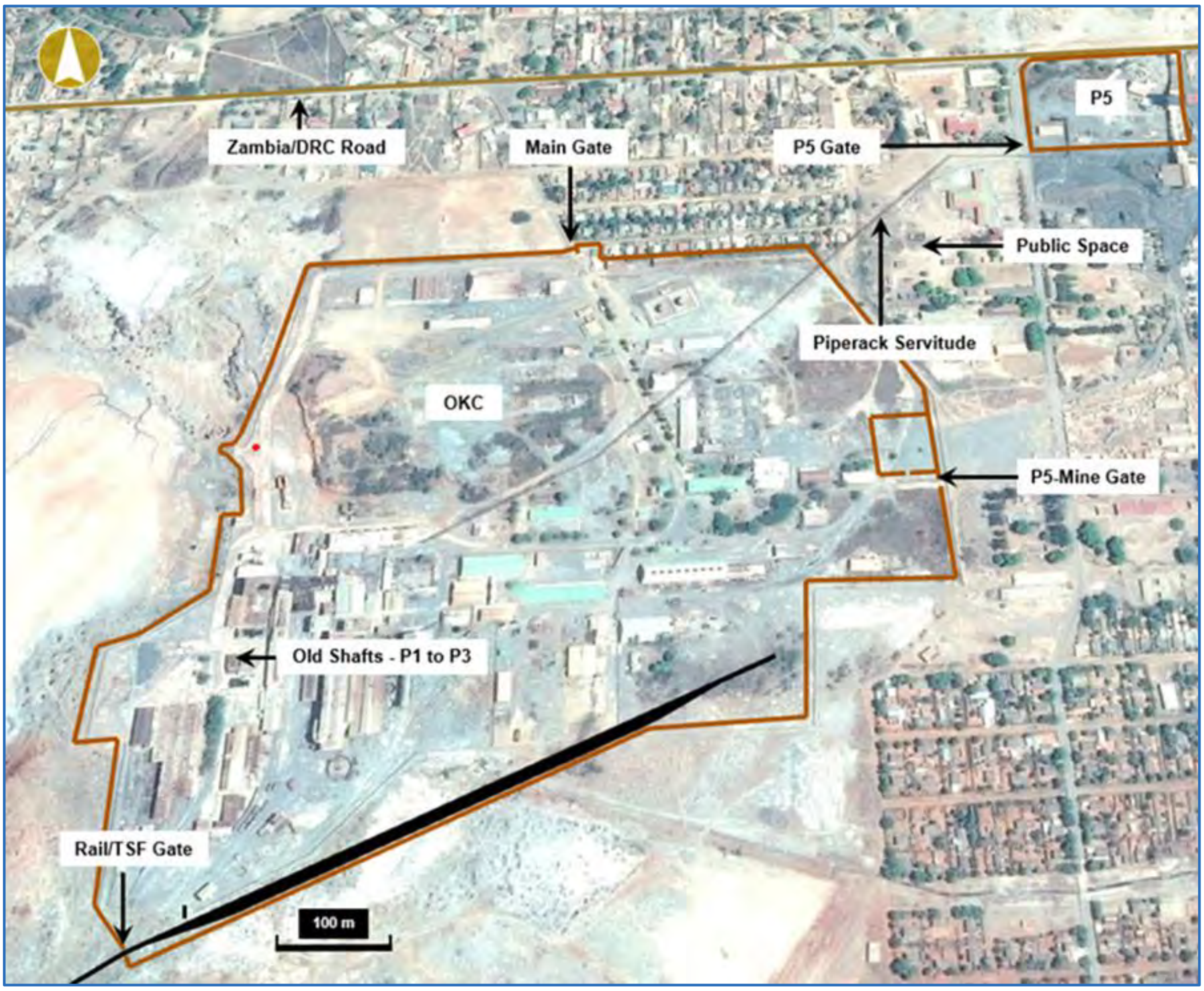
18.2 Access Infrastructure

Apart from the specific mining and plant site areas, the interconnection upgrades are limited. The pipe rack will be replaced with a combined conveyor/pipe rack within a fenced servitude. All roads to and within the Project area are of black top construction. Therefore, with the exception of the Tailing Storage Facility (TSF) access road, no new access infrastructure or upgrades are required for the project. Regional access to Kipushi and local access to the mine is illustrated in Figure 18.2 and Figure 18.3 respectively.

Figure 18.2 Local Access

MDM, 2017

Figure 18.3 Mine Access



MDM, 2017

18.3 Earthworks

Earthworks and terracing requirements were based on an engineering geotechnical investigation undertaken by SRK Consulting (South Africa) (Pty) Ltd (SRK). The work included 13 rotary core bore holes, the excavation of 62 test pits at various areas of the mine as well as Drop Cone Penetrometer (DCP) testing adjacent to test pit positions. Selected soil samples retrieved from the bore holes and test pits were submitted to Specialised Testing Laboratory and Geostrada laboratories in South Africa for geotechnical testing.

The geotechnical investigation showed that:

- The in-situ materials generally classify as soft materials down to a depth of 1.75 m.
- The colluvial and residual clayey and silty soils across the site are not suitable for engineered fills, but are suitable for bulk fills.
- The high clay content of the soils will result in trafficability problems after rainfall.

- Where the soils exhibit voiding by termites, they must be treated to remove the collapse potential prior to construction.
- The design of the foundations was based on the findings and recommendations as published in the SRK Report No 531555/Plant/Final date April 2020.
- As hard material generally occurs below three metres depth, no allowance has been made for removal of rock.

18.4 Roads

New roads required for the project are designed to link up to the existing roads on the mine and will all be finished with a gravel wearing course. Engineering for road earthworks is subject to the same geotechnical considerations as those stated for terracing.

18.5 Weighbridge

Two truck weighbridges have been provided to measure the gross and empty weights of trucks entering and leaving site. The weighbridges are 4.5 m W x 25.65 m L and will accommodate a truck with a Gross Vehicle Mass of 80 t.

18.6 Mobile Equipment

The plant and infrastructure mobile equipment list was developed to meet project requirements. The mobile equipment presented relates to equipment provided by KICO for plant and infrastructure operations, including third party service providers such as: the laboratory contractor; the cleaning, catering and laundry contractor and the security service contractor. The TSF and backfill operations will be outsourced.

18.7 Electrical Infrastructure

18.7.1 Electrical Power Supply and Switch Yard

Power is supplied by SNEL in the DRC. The Kipushi Mine, is connected to the national electrical grid through two power lines, one at 110 kV and the second at 50 kV. Both power lines are equipped with copper conductors and exposed to frequent incidents of conductors' theft. A third power line built in the nineties was vandalised. To mitigate the risk of flooding the mine in case of a prolonged power supply interruption, a project aimed at repairing this line and stringing it with aluminium conductors has been initiated. The scope will also cover the replacement of copper conductors on the existing 110 kV with aluminium conductors and modernising the equipment at both Lubumbashi RS and Kipushi terminal substations.

The incoming lines feed t416ere transformer bays (2 x 110/6.6 kV and 1 x 50/6.6 kV), adjacent to the outdoor yard. All three transformers can operate in parallel. This switchyard is in a reasonable condition and does not require any upgrade for the project.

There are three existing substations:

- Main Substation adjoining the existing switchyard and transformer facility.
- Shaft 5 Substation at the shaft.
- Cascades Substation provides power to Shafts 1–4.

All these substations will continue to be used and costs have yet to be determined for the refurbishment thereof.

18.7.2 Transmission and Distribution

The mine has an extensive system of underground tunnels that are used to distribute power from the main switchyard to Shaft 5 and to the OKC. These tunnels are all operable and will continue to be used going forward.

A 2.25 km, 6.6 kV overhead line will be installed to provide power to the TSF. The overhead line will incorporate an optical ground wire. This wire serves to provide both a grounding and communications function between the TSF and the plant.

18.7.3 Security of Supply and Emergency Power

An outage schedule for 2016 was provided by KICO. For the year in question, power availability was high at 99.58% (37 hours of down time for the year). In moving forward, the following points should be noted:

- The Kipushi Mine has 4 MW of installed back-up power, supplied by two diesel generators (1 x 1 MW and 1 x 3 MW). This generation capacity was not designed to run either the mine, plant, or the dewatering systems independently from the grid. It will, however, run ventilation fans and the shaft hoist in an emergency.
- New 14 MW of back-up power in the form of diesel generators will be provided on site to enable all other critical operations such as underground pumping, to continue in times of power outages.
- A low voltage emergency power section has been allowed for in the process plant MCC at 525 V. This emergency section is to feed dedicated emergency loads within the process plant section.

18.7.4 6.6 kV Switchboard

A new secondary 6.6 kV single busbar switchboard will be provided for the new process plant and will be housed in the process plant centralised substation. This new switchboard will be fed from the existing 6.6 kV Main Substation.

18.7.5 Low Voltage Reticulation

Secondary distribution is at 525 V. Star points of the distribution transformers will be resistively connected to earth. Transformer secondaries shall be rated at 550 V to allow for volt drop at full load.

18.7.6 MCC Substations

Generally, the process plant motor control centres (MCCs) will be housed in the centralised, brick built, low voltage plant substation. Remote MCC's will be housed in steel substations that are converted 12 m high-cube shipping containers. The containers have been insulated and fitted with a tropical roof. All substations are elevated for cable access.

18.7.7 Transformers

Electrical loads are allocated to MCCs, and associated transformers. These loads are grouped by process areas as far as practicable, considering transformer loading and voltage regulation. The MCC designs have been based on 2,500 kVA transformers. Distribution transformers are 6,600/550 V, vector group Dyn11.

Infrastructure lighting and small power would be fed from 6,600/420 V mini-substations, while plant lighting and small power will be fed from dedicated 525/420 V transformers.

A transformer loading schedule has been completed for each transformer.

18.7.8 Motor Control Centres (MCCs)

The MCCs will be of steel construction, free standing, bottom cable entry and either front access and operation for containerised MCC's or back-to-back configuration for the MCC's housed in the plant substation. All MCC's will be adhere to a fully compartmentalised design (form 3b and 4a). The operating voltage will be 525 V, 50 Hz with a control voltage of 110 V, 50 Hz supplied from an internal control transformer. The design fault level will be 50 kA at 525 V for all transformer fed MCC's. A power metres will be provided per MCC incomer and connected to the plant supervisory network.

Motor starters will be direct-on-line (DOL) where motor kW is less than or equal to 90 kW, unless otherwise specified as a Variable Speed Drive (VSD) by process requirements. Motors above 90 kW will be started by Soft Starter or VSD depending on the application. DOL starters are typically equipped with a triple pole, Moulded Case Circuit Breakers (MCCB), contactor (Type 2 coordination) and overload relay(s). Contactors will be rated for AC-3 duty.

All soft-starter / variable speed drive (VSD), starting and stopping and status information will be communicated over the Ethernet network to the PLC / SCADA system. All DOL starting and stopping, and status information will be communicated via localised MCC input/output (I/O) modules to the PLC/SCADA system.

18.7.9 Field Equipment

All drives will be equipped with local isolator E-stop stations with latching e-stop. These will be field mounted within robust steel drip covers. Plant start-up sirens will provide a warning for conveyor and large equipment drives about to start. All emergency functions such as emergency stops are to be hard wired, but will also be monitored by the PLC.

Generally, VSD's will be mounted within the MCC, large VSDs, mounted external to the MCC as standalone cubicles.

18.7.10 Motors

Low voltage (525 V) motors will be designed to IEC 60034, for continuous duty class S1. Insulation will be Class H. Temperature rise will be limited to 80°C (Class B). Enclosures will be IP55 to IEC 60034-5. Premium efficiency (IE3) motors will be used throughout the design and shall be of the totally enclosed fan cooling type (TEFC).

6.6 kV Motors will include Zorc Surge suppressors that will be fitted separate to the motor cable box (MV motors). Temperature monitoring devices are to be fitted to bearings and windings on MV motors.

18.7.11 Cable

Medium voltage (6.6 kV) cables will be individually screened copper conductor three core XLPE/PVC/SWA/PVC 6.35/11 kV cable to SANS 1339. Single core cables will be of XLPE/PVC/SWA/PVC construction and be arranged in prescribed trefoil formation (gland plates will be of non-ferrous construction).

Low voltage cables will be copper conductor PVC/PVC/SWA/PVC 600–1,000 V cable to SANS 1507. Standard flame-retardant cable is to be utilised for surface installations. Power cables shall have four cores, the fourth core being utilised as an effective earth between the equipment (e.g., motor) and the substation earth bar.

Conductor sizes for 525 V motor feeders shall be sized to ensure reliable motor starting. Cables are sized for a maximum 5% voltage drop during full load condition. Start-up voltage drops are determined on a case-by-case basis based on the starting torque requirements.

18.7.12 Cable Racks

The preference is for cables to be mounted either in the underground cable tunnels on site, or above ground level on suitable cable racks or overhead line systems. Where necessary, buried cables will be in trenches and will be provided with cable markers on surface at 10 m intervals and changes in direction as per electrical installation specification. Cable trenches will be backfilled with a suitable material to ensure effective heat transfer from the cable to the surrounding earth. A detailed services servitude plan is to be made and kept up to date.

18.7.13 Earthing

Detailed earthing and lightning protection design will be carried out for new areas. The earthing values shall be in accordance with SANS 10198, SANS 10199, SANS 10313.

Earth reading values of less than 10 ohm shall apply for general plant structures and conveyors. All electrical equipment shall be earth bonded via the substation earth bar.

The various earths shall be linked using buried 70 mm² bare copper earth wire.

18.7.14 Lighting and Small Power

The lighting and small power design shall generally comply with the provisions of the South African Occupational Health and Safety Act. Lighting will be by a combination of fluorescent, LED bulkhead and LED floodlights to achieve illumination levels required. The lighting levels are detailed in the Electrical Design Criteria (EDC).

Emergency lighting has been allowed for in key areas. Emergency lights will be fed from dedicated UPS circuits. Photoelectric switches will control the exterior lights. Provision has been made for weatherproof 230 V 16 A switched socket outlets and 525 V 63 A welding socket outlets.

18.8 Water Management, Supply and Distribution

18.8.1 Overview

There are four primary sources of water on site, namely:

- Town and potable water from a spring,
- Underground water from Shaft 5,
- Underground water from Shaft 3, and
- Water pumped from the pit.

With respect to water management on site, it is relevant to note that:

- Raw water for the plant and for surface infrastructure is sourced from the underground workings.
- Water is not recovered from the TSF or from site drains.
- Underground water, which meets the DRC discharge requirements is discharged directly to the site stormwater drains.
- The mine largely falls in one catchment area (southern catchment area), with water running west–east and north–south-east to the Kipushi River via tailings dam 3.

18.8.2 Potable Water

Potable water for the mine and the town has been reviewed by Golder and costs have been provided to KICO for the refurbishment of the town's water supply system for incorporation in the Kipushi 2022 FS estimate.

- Supply:
  - Number of boreholes: 10
  - Volumetric flowrate required: 5 ML/d (excluding mine requirements)
- Storage:
  - Northern water tower: 3.5 ML
  - Southern water tower: 2.0 ML

- Distribution (new potable water tank): 1.27 ML/d
  - Process plant nominal flow: 0.912 ML/d (process and reagent make-up)
  - Mine potable water (proposed): 0.24 ML/d
  - CRF Facility: 0.12 ML/d

18.8.3 Raw Water Supply to Mine

Raw water is pumped from P5 Shaft into the Raw water tank with excess water being discharged into the North cut off drain as it meets the DRC discharge requirements.

The Kipushi 2022 FS has allowed for:

- A new raw water tank has been allowed for at the intersection of conveyors T6 and T7. The raw water tank services the firewater requirements for the process plant as well with the design having a dedicated volume for fire suppression services.
- Replacing all steel water pipes between Shaft 5, Shaft 3, and the water tower with equivalent sized pipes fabricated from High Density Polyethylene (HDPE).
- Installing piping for conveyor and stockpile dust suppression systems.

User of underground water and the approximate quantity of water used are defined below:

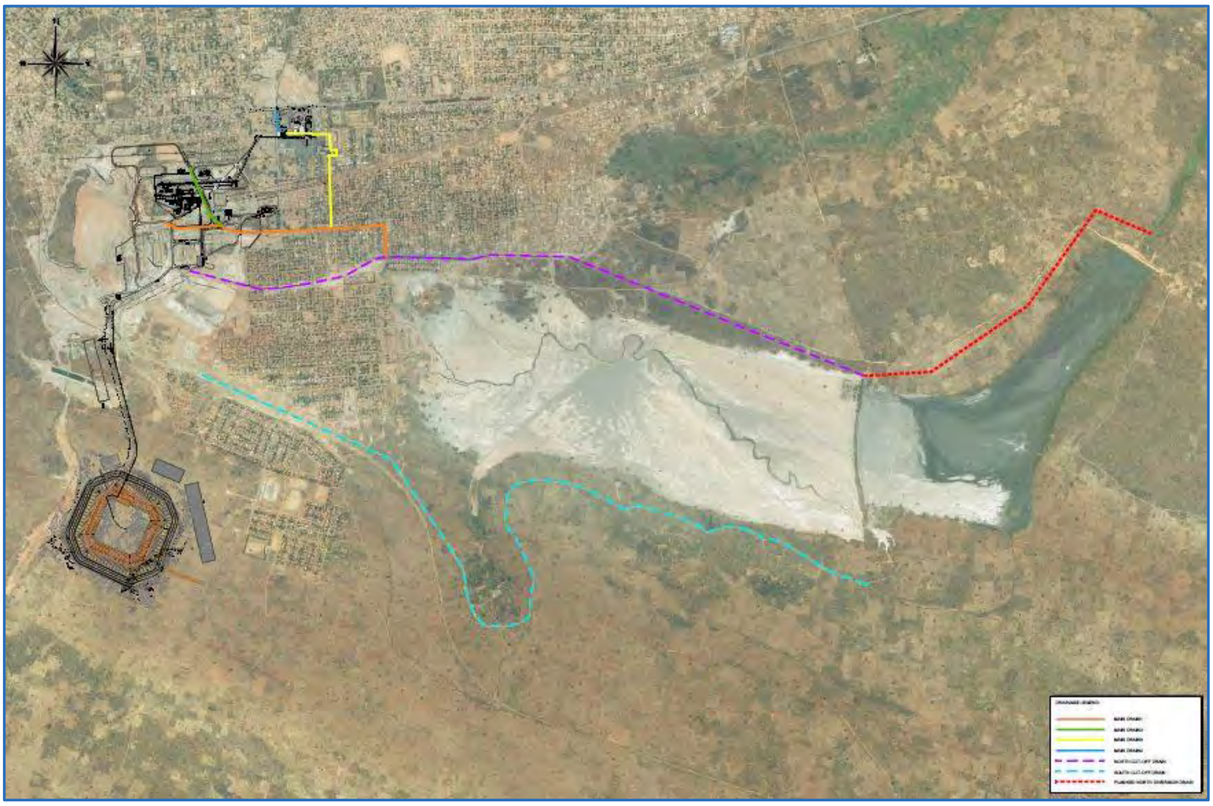
- The plant will require 16.3 m^3/h of underground make-up water, if the water from concentrate and tails thickeners are returned to the plant, if not raw water make-up will increase to 26 m^3/h.
- Vehicle workshops (wash bays) and the fixed dust suppression system will require 10 m^3/h of underground water.
- Dust suppression requirements (mobile equipment).
- Fire water system (ad hoc user).

18.8.4 Stormwater Management

There are two catchment areas associated with the mine, namely the northern catchment area that runs to the north of the road between Shaft 5 and Lake Kamalenge and the southern catchment area, which runs to the south of the aforesaid road and drains to the Kipushi River via TSF 3. The sites drainage system is highlighted in Figure 18.4.

For the Kipushi 2022 FS, only the southern catchment area is used. Given that the drains within METC's scope of work are currently being used and are in a reasonable condition, no capital cost allowance has been provided for stormwater drainage systems. Rather, an ongoing maintenance function and budget has been allowed for in the operating cost estimate for cleaning and repairs. The premise that the South catchment drain (green line) is not relevant to the projects.

Figure 18.4 Kipushi Mine Drainage System



METC, 2022

18.8.5 Discharge of Water from Mine Site

Currently, all water falling on the site and/or emanating from underground meets the DRC Government's discharge requirements and is discharged directly into the environment without treatment or containment.

18.9 Public Health Services

The FS has allowed for the construction of a new sewage treatment plant at the metallurgical process plant site. Ablution facilities around the process plant will drain into the new facility. Sewage from P5 shaft and other facilities will collect in septic tanks from where it will be collected and trucked to the sewage treatment plant for treatment.

18.10 Fire and Emergency Services

An existing building and currently in use will be upgraded to provide fire and emergency services facilities to house emergency personnel, equipment, and emergency vehicles.

18.11 Fire Protection

Fixed fire-fighting infrastructure will be installed to supplement the mine's mobile fire-fighting capability. Fire water will be drawn from the plants raw water system (underground water), and allowance have been made for provision of new fire water pumps and reticulation to both new and old buildings.

Fire-fighting apparatus such as hose reels have been estimated for inclusion in the new and refurbished buildings. For conveyors, installation of fire hose reels have been allowed for at specific distances and fire extinguishers have been provided at each drive, take-up, and tail and transfer tower.

Fire detection will be included in new LV substations with the present MV substations using their existing fire detection systems which are assumed to be adequate. Handheld fire extinguishers will be placed in and around each new LV substation for firefighting purposes.

In addition to the fixed fire-fighting equipment provided, one fire tender has been allowed for in the estimate. Water for the operation of this equipment will be sourced from either the fire water system or mobile dust suppression tankers (when working remotely from fixed infrastructure).

18.12 Fuel and Lubricant Supply

New diesel storage facilities will be established for the project, two 65 m^3 self-bunded tank facilities will be located at the emergency generator plant and a third 65 m^3 self-bunded tank will be positioned at the P5 shaft. Dispensing facilities for light vehicles will be provided at the emergency power facility and plant mobile equipment will be refuelled utilising a diesel bowser.

Lubricants delivered, will be stored in a new 84 m^2 portal frame structure, whilst waste oils will be stored in an adjacent 84 m^2 portal frame structure. Both facilities are located on a concrete bunded slab.

18.13 Communications and IT

Currently on site, KICO communicates internally and externally using:

- A satellite connection (c-band),
- Combined fibre (5 MBps) and line of site radio connection mounted on the Shaft 5 tower,
- Cellular connection (Vodacom),
- Television (DSTV or equivalent connection),
- Radio, and
- Wireless connection on site and in the guesthouses.

Refurbishing of the existing facilities have been done within the historic operating budget of the Kipushi facility.

- The Kipushi 2022 FS has made allowance for the following additional infrastructure: Plant and process control data, security and general communication system data will be run on separate networks.
- The TSF and the main site will be connected using an Optical Ground Wire (OPGW) on the overhead power line.

IT hardware and software (including specialist software) have been allowed for in each of the areas and departments.

18.14 Waste Management

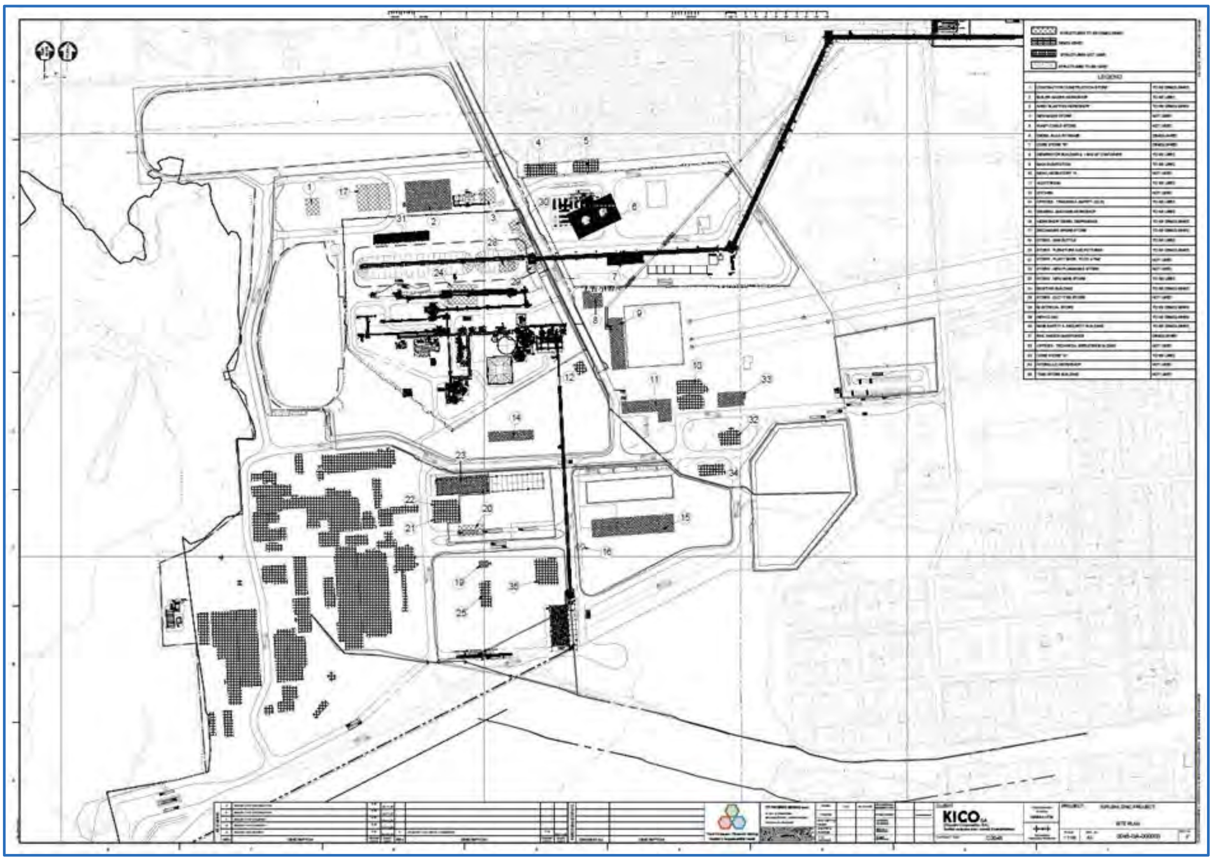
The disposal requirements will be undertaken in accordance with the Project Environmental and Social Management Plan (ESMP). The waste management facilities will include:

- Storage of Waste Petroleum Products,
- Storage of Scrap and/or Recyclable Products,
- Off-site Septic Tank Sludge Disposal,
- Storage of Waste Prior to Landfill,
- Off-site Landfill Disposal, and
- Incinerator.

18.15 Buildings and Structures

As far as practically possible, old buildings and structures will be refurbished and, in some cases, repurposed to meet project requirements. Where necessary, new buildings and structures have been allowed for. Given the relatively short mine life, the approach has been to ensure that building refurbishment is fit for purpose and new buildings are either of the portal frame, prefabricated or containerised types. These are shown in Figure 18.5.

Figure 18.5 Kipushi 2022 FS Building, Structures, and Rooms



METC, 2022

18.16 Laboratory

Whilst there is an existing partially functioning laboratory on site, comprising of a 350 m^2 analytical lab and a 320 m^2 sample preparation lab, it was decided to not refurbish these facilities on the basis that the sample preparation laboratory is not suitable for the new duty and the cost of refurbishing the existing facilities to meet the new project demands were too similar.

A new portal frame and containerised laboratory with the requisite equipment have been allowed for, as described more fully below.

The laboratory facility includes:

- Bulk sample preparation laboratory
- Analytical laboratory
- Laboratory information management system.

18.17 Workshops

Historically the site was largely self-reliant with respect for the maintenance of equipment. The business model employed was one of a large central site workshop, supported by a number of smaller workshops in different geographic and business areas. It is planned to consolidate the workshops and the remote workshops be re-purposed.

The workshops planned are:

- General Machine Workshop: existing building for mechanical and machine workshop, hydraulic workshop, electrical and instrumentation workshop.
- Welding Workshop: existing building for site welding activities.
- Light and Heavy Vehicle Workshop: existing building for maintenance of mobile equipment, tyre changing, fuel and lubrication and included in the refit will be new cranage.
- Vehicle wash down bay: a new facility of packaged washing equipment, tanks and, pumps including waste management controls.

18.18 Stores and Construction Laydown

The store buildings and laydown areas will be a mix of new and existing facilities. The key facilities are:

- Mine Light Equipment Store: a new facility using an existing building with laydown area.
- Flammable Stores: a new facility using an existing building. A sperate store is planned for gas bottles.
- Concentrate Bags Store: a new facility with additional outdoor storage yard to cater for concentrate transport interruptions.
- Reagent Make-up and Storage Area: a new building.
- Laydown Areas: continue with existing areas, new gate houses will be installed.

18.19 Security and Access Control

The main access control points for the site are:

- Main gate (northern entrance),
- Shaft 5 gate house (western entrance),
- Main Mineshaft 5 gate house (eastern entrance),
- Shaft 5 main road gate (northern entrance),
- TSF road gates (southern entrance), and
- TSF gates.

The fencing is summarised below:

- Type (existing): ClearVu (www.clearvucom), 3 m high with spikes

- Total fence line: 8,211 m
- Shaft 5 and Plant – New fencing required: 2,897 m
- Fencing to be taken down, moved and re-instated: 700 m
- TSF fencing: 1,840 m

Cameras have been allowed for on the Kipushi Mine connected to the security workstations via the site's fibre optic network. Access control systems comprising of manual and automated gates, turn-styles, personnel readers, and linked CCTV systems have been provided for in the cost estimate, along with the associated time and attendance software, badges, and cards.

18.20 Accommodation

Accommodation is provided in the town of Kipushi to personnel and to some mine subcontractors. No housing or offices have been allowed for in Lubumbashi. Accommodation costs are planned to be included in the contractor costs. No accommodation have been provided for construction personnel, on the basis that:

- The appointed earthworks, civils, and SMPP contractors will be local to Lubumbashi.
- Personnel associated with vendors and the E&I and EPCM contractor, will be accommodated in hotels or guest houses locally.

The company does not own accommodation in Kipushi, but rather refurbishes and leases houses in the town.

For the delivery of accommodation services, a third-party service provider will be employed to provide a centralised mess at the main guest house and a laundry function at the mine.

18.21 Concentrate Transport and Logistics

18.21.1 Transport and Logistics Summary

Given the saturated roads and border crossings, a sustainable logistics solution for Kipushi is critical for the viability of the mine project and continued stability of existing freight flows in and out of the Copperbelt.

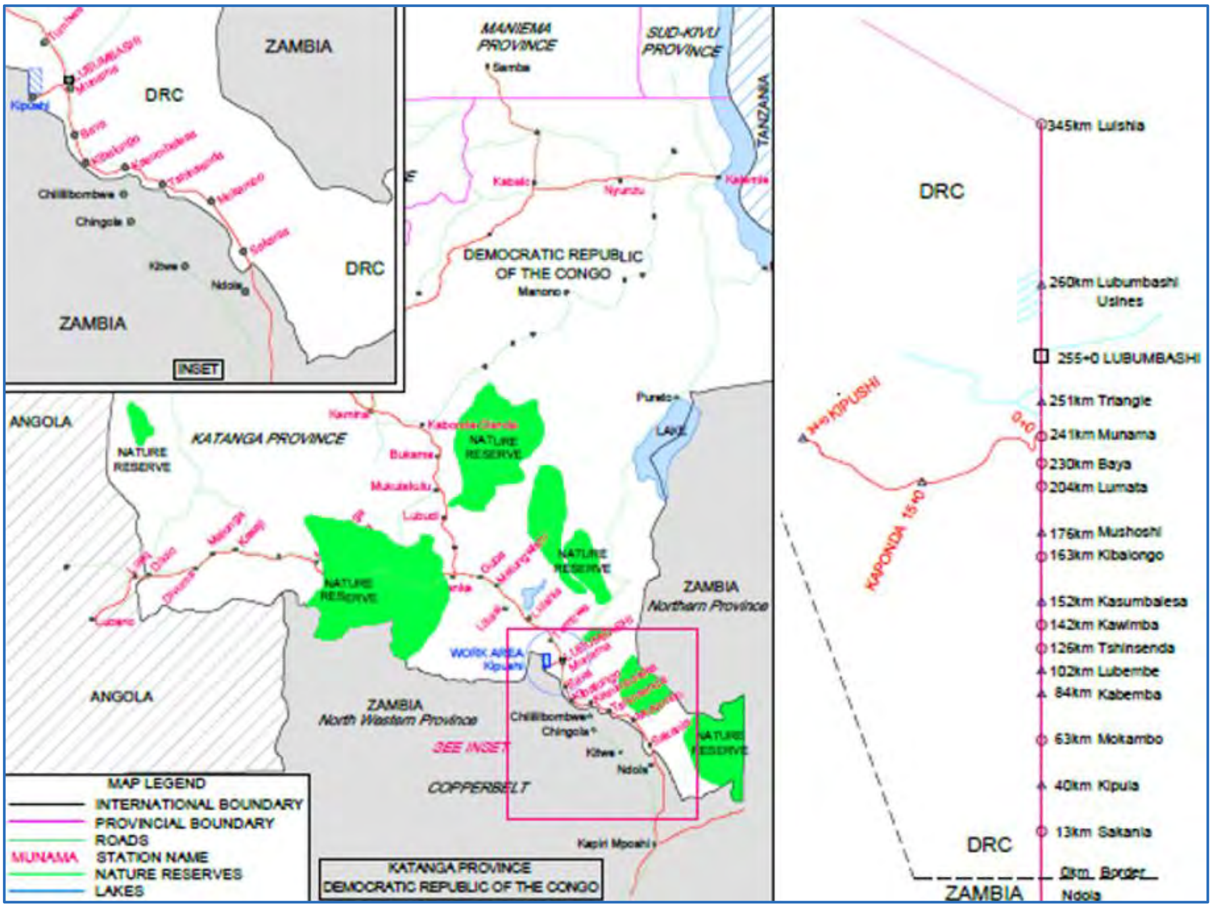
From Kipushi to an ocean seaport there are various established road corridors within the Southern Africa Development Community (SADC) region. All of these routes are supported and promoted by the SADC Secretariat as part of their regional trade development commitment, and harmonisation of Customs border procedures are an ongoing process within the region.

Location and connections from the Kipushi Mine are shown in Figure 18.6.

As part of the PFS a series of intermediate stages linking transport operations from the mine to the customer were evaluated. These stages comprise roads, railway lines, and shipping routes in addition to intermediate terminal handling options, whether in a single or multimodal logistics choice. 'Most economical' typically refers to the optimal selection and combination of logistic elements such as, packaging type, materials handling, transportation, and routes, all of which contribute to a competitive overall logistics cost and service. In this case, the supply chain options included combinations of the following elements:

- Transportation mode (road and rail).
- Wagon type (bulk and flatbed).
- Packing type (bulk, bagged/break bulk, containerised).
- Intermodal terminals and transhipment.
- Return loads in order to subsidise the export costs, so-called backhaul dividend.
- Service-related costs such as delays due to inefficient inter-country operations fluidity.

Figure 18.6 Location and Connections from the Kipushi Mine



R&H Rail, 2019

A total of 18 supply chains, comprising 160 options, based on permutations of packaging type, port of export and final destination, mode of transport, and the use of intermodal facilities were developed and evaluated to identify the most economical option.

The PFS identified rail as being the preferred option for transporting zinc concentrate from the mine to a port of loading.

The FS developed alternate supply chain options, bypassing the rail network, and evaluated the associated costs and benefits according to the very same criteria as in the previous study. The new supply chain options included:

- A direct road-only supply chain from the mine to the port, utilising the existing Kasumbalesa border.
- A multimodal (road to rail) supply chain, whereby cargo is road hauled from the mine, through the Kasumbalesa border, to Ndola (118 km). Thereafter, the cargo is transhipped onto rail for further conveyance to the port.
- A multimodal (road to rail) supply chain, utilising an existing road via Solwezi in Zambia, which is undergoing refurbishment. Cargo is road hauled from the mine, through the Kipushi border, via Solwezi to Ndola. Thereafter, the cargo is transhipped onto rail for further conveyance to the port. The road originates at the mine and links Zambian T5 network near Kansanshi mine. The Kipushi border currently facilitates light vehicles. This option would require the Kipushi border to be upgraded to facilitate mineral exports.

The outcome of the options were compared with the direct rail supply chain from Kipushi to each port (excluding Walvis Bay), identified in the previous study as the preferred mode of transport and is used as a benchmark against the new supply chain options. The major risk associated with this option being the poor performance of the rail network.

The Kipushi 2022 FS evaluation found that currently the most favourable option was to transport direct to port via road only.

The study has assumed truck haulage direct to port as break bulk concentrate out of either Durban, Walvis Bay or Dar es Salaam to China (Shanghai).

18.21.2 Transport Options

Both rail and road concentrate transport options were reviewed. There are various established road corridors from Kipushi to an ocean seaport within the Southern Africa Development Community (SADC) region. All of these routes are supported and promoted by the SADC.

The base case for the Kipushi 2022 FS is a road transport option via Solwezi to Ndola and then direct to port as break bulk concentrate out of either Durban, Walvis Bay or Dar es Salaam to China (Shanghai).

The direct rail supply chain from Kipushi to each port (excluding Walvis Bay), identified in the previous study as the preferred mode of transport was excluded due to the major risk associated with the poor performance of the rail network.

The principal risk for the road haul options from Kipushi to Impala Terminals' intermodal facility on the Likasi Road is that the proposed Impala loading facility on the Likasi Road north of Lubumbashi is currently only partly constructed and has been mothballed pending new investment.

The Kipushi 2022 FS has made allowance for upgrading of the border post dedicated to Kipushi, which includes a truck staging area complete with ablution facilities.

18.21.3 Multimodal (road to rail) Transport

The multimodal transport option involves road transport via Solwezi to Ndola (421 km) where the concentrate is trans-loaded onto rail, linking up with the existing North–South Rail Corridor. The option requires refurbishing an existing road from the Kipushi border, which links to the Zambian T5, near Solwezi. The current border post only facilitates light vehicles and will require capital to upgrade the infrastructure to accommodate heavy vehicles. The intention of upgrading this border post and the road network is to develop a dedicated border post for the Kipushi Mine, ultimately mitigating the excessive dwell time and high demurrage charges imposed at Kasumbalesa. However, based on the funding model and government intervention, if a dedicated border is unachievable, there is a risk that the existing queues at Kasumbalesa will overflow to the Kipushi border. This could result in a shorter dwell time than currently experienced at Kasumbalesa, but may not prove to be beneficial as the road haulage distance to Ndola is doubled, resulting in higher haulage costs.

18.21.4 Current Rail Corridors

The railway connection from South Africa to the Copperbelt is today vastly underutilised, with carried annual transit freight volumes of only 288 kt in 2016 compared to its current capacity of around 3 Mt. Figure 18.7 details the SADC Rail Network. The full rail infrastructure route from Ndola to Durban is operational and any problems arising from sections of track in poor condition are overcome by running trains at slower speeds. These slower speeds are offset by night operation of trains (whereas many road trucks cannot move in darkness) and the much faster clearance of rail wagons at international borders (two hours in most cases as goods travel in bond). The resultant average speed of a train on the North–South Rail Corridor (NSRC) in 2016 was about 16 km/h. In comparison to road, with night travel by rail and minimised border delays, the journey from Lubumbashi to Durban can be achieved by rail in 200 h, or nine days which is as fast as currently achieved by an average road convoy.

In the first quarter of 2017 the NSRC operated approximately thirty trains every day along the full length of the corridor. Approximately 90% of the corridor's capacity is currently unused. KICO would require under two trains per day from Ndola.

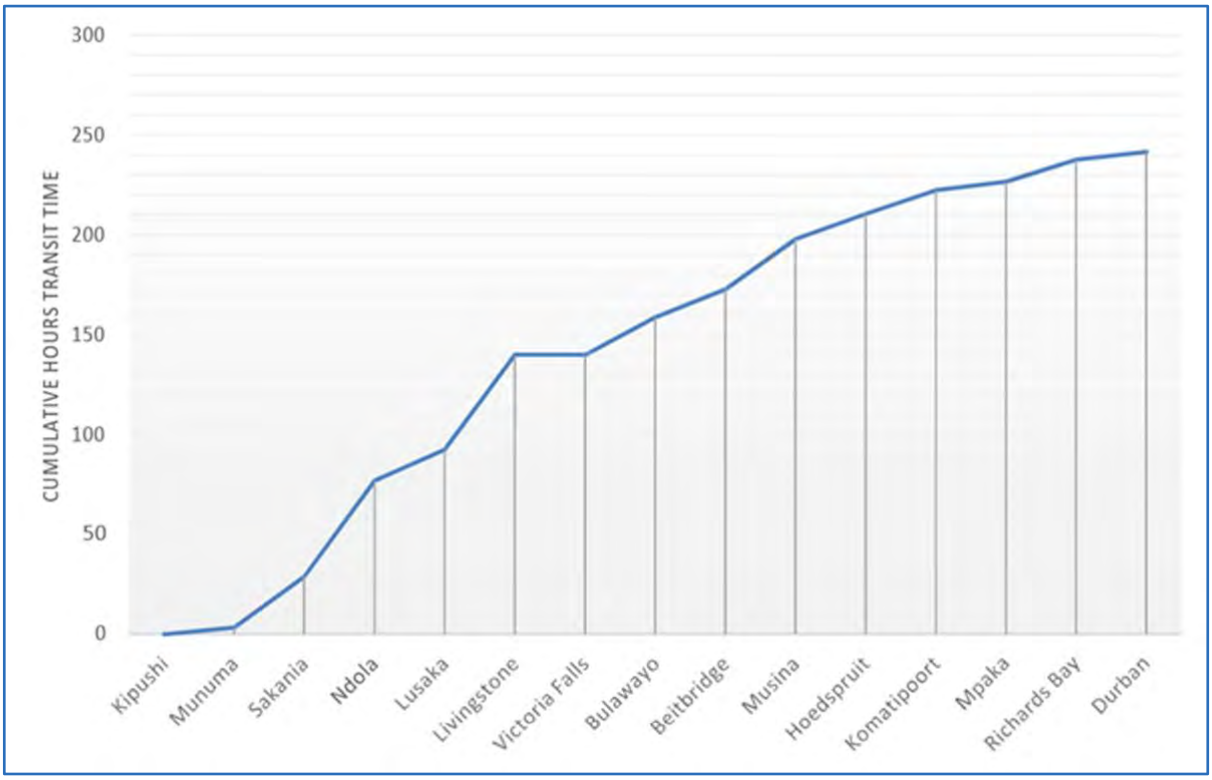
Figure 18.7 SADC Rail Network

Grindrod, 2016

18.21.5 Rail Operation Plan

For the Ndola to Durban rail journey, a transit time of approximately seven days is anticipated broken down by sector shown in Figure 18.8.

Figure 18.8 Train Service Chart – Kipushi to Durban



Grindrod, 2017

18.21.6 Ocean Shipping and Freight Rates

The Kipushi 2022 FS considered the merits and disadvantages of transporting zinc concentrate from Kipushi in bulk and bagged modes or containerising inland at the mine. The concentrate is then to be shipped out of Durban to China (Shanghai).

The analysis shows that a break bulk solution would be the most cost effective whereby concentrate is prepared for shipment in 'big bags' of up to 2.2 t each at the mine and hauled on rail in open box wagons to a terminal near Durban or in Richards Bay.

For shipment parcel sizes of up to 5,000 t bags would be packed 10 to a box inside a standard 20 foot shipping container. This containerised solution would allow the project to take advantage of cheap backhaul container shipping rates out of Durban to the Far East.

18.21.7 Existing Road Transport Corridors

Given the already saturated roads and border crossings, a sustainable logistics solution for Kipushi is critical for the viability of the mine project and continued stability of existing freight flows in and out of the Copperbelt.

From Kipushi to an ocean seaport there are various established road corridors within the Southern Africa Development Community (SADC) region. All of these routes are supported and promoted by the SADC Secretariat as part of their regional trade development commitment, and harmonisation of customs border procedures is an ongoing process within the region.

It has been reported that substantial progress have been made in customs processes at international borders, as road haulage freight have increased, most main road arteries in the region are seriously congested, and traffic at border crossings often takes days rather than hours to clear. Figure 18.9 to Figure 18.13 show the following road routes from Kipushi to various ports:

- Kipushi to Durban via Road (2,716 km, three border crossings).
- Kipushi to Richards Bay via Road (2,604 km, three border crossings).
- Kipushi to Maputo via Road (2,300 km, four border crossings).
- Kipushi to Beira via Road (1,605 km, three border crossings).
- Kipushi to Dar Es Salaam via Road (2,039 km, two border crossings).

Figure 18.9 Kipushi to Durban via Road

Grindrod, 2017

Figure 18.10 Kipushi to Richards Bay via Road

Grindrod, 2017

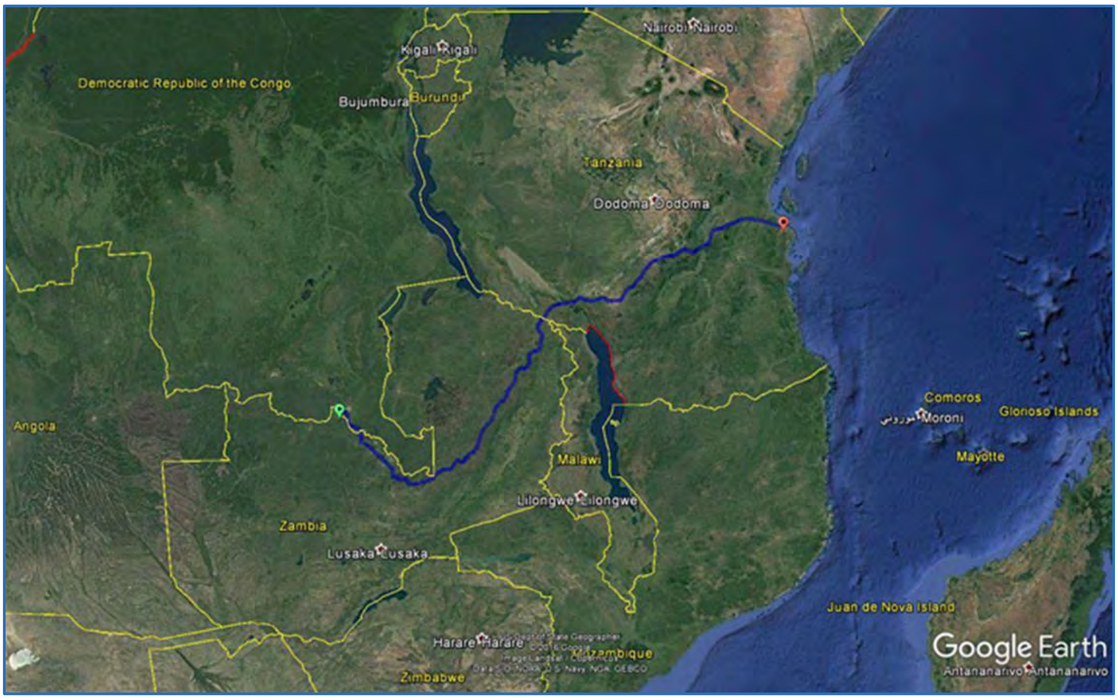
Figure 18.11 Kipushi to Maputo via Road

Grindrod, 2017

Figure 18.12 Kipushi to Beira via Road

Grindrod, 2017

Figure 18.13 Kipushi to Dar Es Salaam via Road



Grindrod, 2017

18.21.8 Insurance and Inspection

Ocean marine cargo insurance can be obtained for all concentrates shipped by vessel. Under CIF contracts, marine insurance is taken out by the seller in the name of the buyer in the amount of 110% of the estimated value of the concentrates in each shipment. Risk of loss, excluding normal handling losses, passes to the buyer as concentrates are progressively loaded onto the carrying vessel. Marine insurance rates typically average around 0.05–0.07% of the estimated invoice value (adjusted to 110%), i.e. the payable metal value, less all treatment and refining charges, as well as any penalties and price participation which may apply (the Net Invoice Value (NIV)).

Inspection services are typically employed at the vessel discharge and at the weighing and sampling procedures to ensure that the Seller's interests with respect to the proper handling of the concentrates at the receiver's facilities are fully respected. There are a number of companies that offer these services.

Where a company representative cannot be available to observe vessel loading (and/or conduct regular site visits to ensure the concentrate is being properly stored and handled), shipper's will frequently have representation at the load port to monitor terminal activities.

18.22 Comments on Section 18

The mine has significant existing infrastructure. Costs have been allowed for refurbishment where required.

The mine is producing a high volume of zinc concentrate product which requires good transport infrastructure to export from site as well the associated importation of various reagents and materials for the process. This covers road transport from site and through ports. Transporting to South Africa for port loading and subsequent shipping, are not identified as a risk.

19 MARKET STUDIES AND CONTRACTS

19.1 Zinc Market Overview

The Kipushi 2022 FS plans for the sale of zinc concentrate. Kipushi Corporation SA (KICO) have undertaken market analysis and engaged with potential customers for the Kipushi zinc concentrate. The conclusions from this work are that the Kipushi zinc concentrate will be saleable into the global zinc market. The global demand for refined zinc (Table 19.1) has grown by close to 1.7 Mt over the past decade. As with most other metals, China has become the largest participant in the market, accounting for approximately 53% of global consumption in 2020, up from approximately 40% a decade ago. Future zinc demand is expected to remain steady with growth at 2–3% in the medium term. The key risk to this outlook remains the strength of global economic growth, and Chinese economic growth in particular.

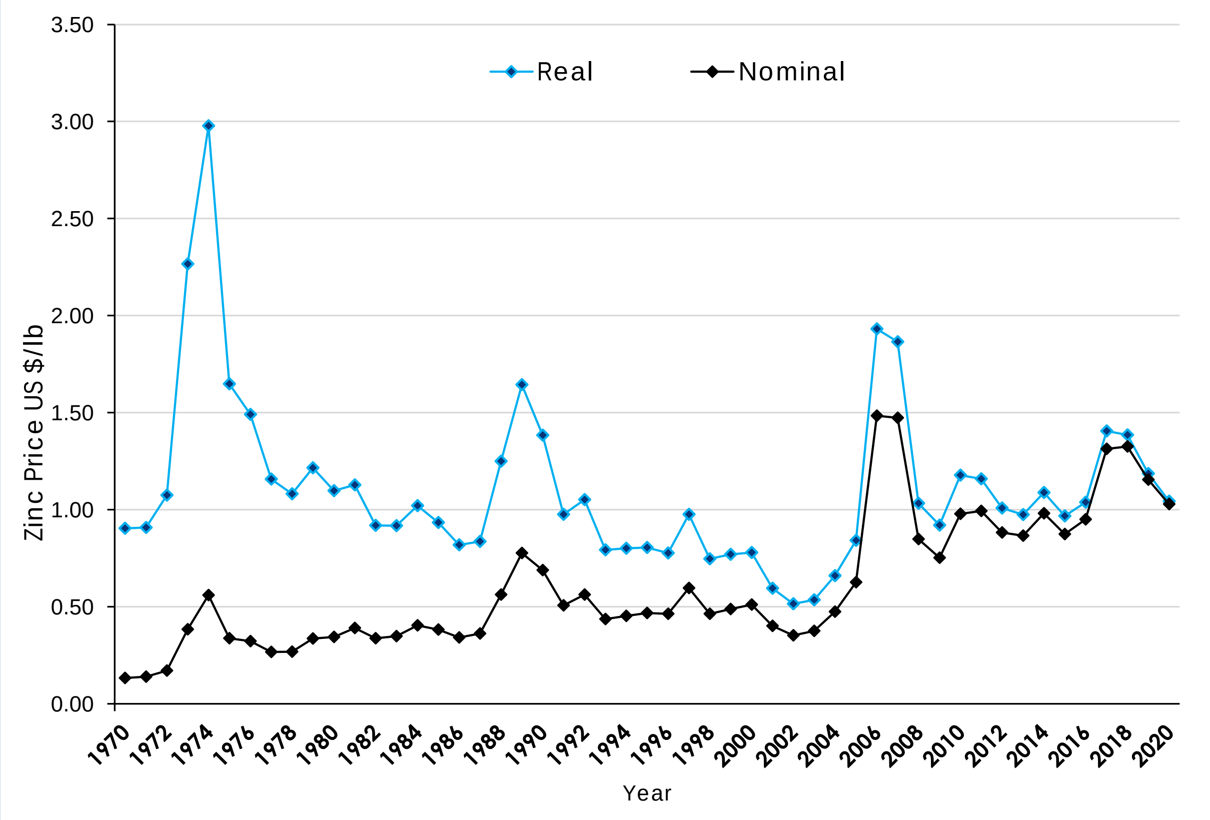
Table 19.1 Global Refined Zinc Supply-Demand Balance

Year	Zn Supply (kt)	Global Demand (kt)	Surplus (Deficit) (kt)
2010	12,720	11,569	1,151
2011	12,965	12,394	571
2012	12,453	12,649	–196
2013	12,935	13,033	–97
2014	13,233	13,508	–276
2015	13,720	13,644	76
2016	13,586	13,947	–361
2017	13,523	14,206	–683
2018	13,247	14,142	–896
2019	13,400	13,868	–468
2020	13,731	13,228	503

Major mine closures from 2016 to 2018 in Australian, Irish, and North American mines have removed production from the market, equivalent to approximately 6% of annual global zinc supply.

Limited investment in new capacity has been attributed to historically poor returns generated by the zinc mining industry where prices trended downward in real terms from the mid-1970s to the middle part of the last decade. During this 20-year period prior to the price spike in 2006 and 2007, the zinc price traded within a wide range of around \$0.27/lb to \$0.97/lb but averaged less than \$0.50/lb (Figure 19.1).

Figure 19.1 Zinc Price (1970–2020)



Wood Mackenzie, 2021

A collective underinvestment in exploration and new zinc mine capacity has contributed to declining mine supply from traditional regions and the current poor development pipeline is expected to affect short, medium, and even long-term zinc supplies. The legacy of this limited investment has been few new significant zinc discoveries. Many of the projects currently in train have been known for many years but have not been developed due to their higher cost structures and/or other challenges (e.g. technical issues, political risk, or lack of infrastructure).

19.1.1 Market Factors

Two major factors could have a bearing on the zinc concentrate market:

- Market Influence of China, and
- Market Consolidation.

China has a significant influence on the zinc market. China is the world's largest producer of zinc; accounting for roughly 37% of global mine zinc production according to International Lead Zinc Study Group (ILZSG) statistics. The Chinese industry is dominated by a multitude of small mines, many of which are reportedly low-grade, running with head grades as low as 3% combined Zn-Pb. Due to their scale and sheer number, it is extremely difficult to quantify actual Chinese production. As the world's largest zinc concentrate producer and as a major concentrate importer, swings in Chinese mine production can significantly influence market balances. Although the pace of expansion in mine output is expected to slow, the potential for ongoing growth could impact the projected world zinc supply contraction scenario.

Urbanisation and industrialisation will remain the dominant driving force behind global zinc consumption. Although the prospects for the developing world economies have deteriorated in recent years, the unstoppable forces of urbanisation and industrialisation mean that in the long term, the developing world will continue to dominate global growth in zinc consumption.

The potential for further zinc industry consolidation may also have a bearing on future concentrate supply. An industry dominated by fewer larger players, each with multiple projects in their portfolio, may contribute to a more disciplined introduction of new mine supply or offer cuts to existing production in an effort to rebalance the market and support prices.

19.1.2 Zinc Smelter Production and Concentrate Demand

The rate of growth of global zinc refining capacity is reported to be slowing and can be attributed to many factors, including:

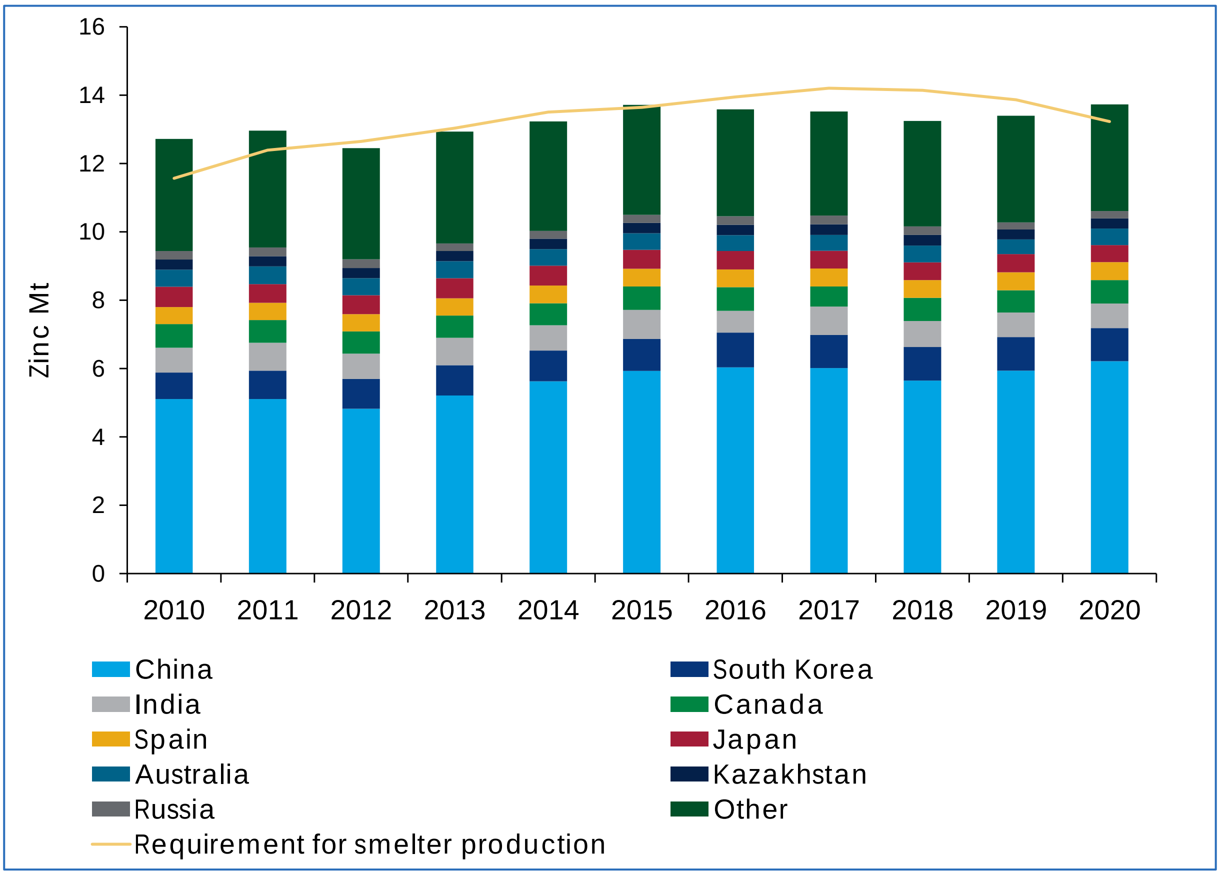
- Reduced profitability due to falling processing charges.
- Concerns about longer term security of concentrate supply.
- Stagnant growth in local metal consumption.
- Rising energy costs.
- Higher capital cost requirements.
- Increasing environmental and social challenges.

Global refined production is still expected to expand, with the majority of the growth expected to continue to come from China.

It is highly unlikely that there will be any greenfield smelter capacity constructed in western countries for the balance of this decade; any new western capacity is expected to be limited to brownfield expansions and debottlenecking.

Over the past decade, in an effort to satisfy growing domestic zinc metal demand, Chinese smelting capacity has increased substantially with a 4.7% increase year on year in 2020, resulting in excess smelter capacity (Figure 19.2).

Figure 19.2 Smelter Capacity



Wood Mackenzie, 2021

Beyond 2022, Chinese smelter capacity is expected to continue grow further with commissioning of several smelter expansions in an attempt to match growing domestic metal demand. However, Chinese smelters are facing increased environmental oversight and therefore, faces several challenges in quickly building smelting capacity. It is speculated that while sufficient zinc refining capacity will be available to meet the demand for metal, mine supply may not meet this demand.

19.1.3 Projected Zinc Concentrate Supply and Demand Balance

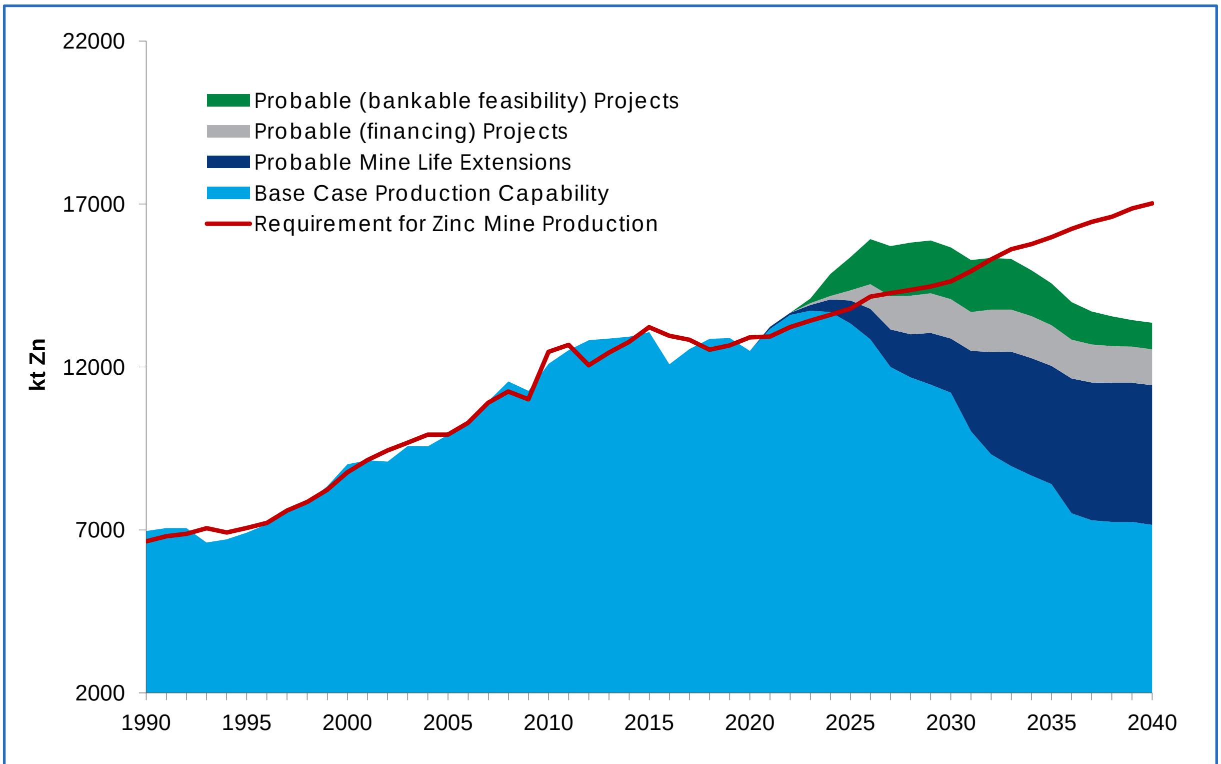
Zinc demand is continuing to be driven by urbanisation and industrialisation of the world economy, which has been led by China as the largest contributor. Significant infrastructure spending in China from 2000 to 2020 has been the main driving force behind global zinc consumption growth of 2.3% per year over this period.

In 2020, the outbreak of COVID-19 resulted in a decline in demand from consumers and businesses, which led to a downturn in global zinc consumption of 4.6% in 2020 compared the previous year. COVID-19 also led to several mine suspensions resulting in approximately 1.4 Mt of production losses and a resulting deficit of 0.4 Mt in the concentrate market, but significant global stocks of zinc concentrate to feed smelters left the market for refined metal in a 0.5 Mt surplus by the end of 2020.

As seen in 2015–2017 with the closure of Century (Australia) and Lisheen (Ireland) mines, and production cuts at Glencore mines, the zinc concentrate market can experience significant short-term volatility resulting from mine supply shocks. Concentrate shortfalls would translate into significantly reduced metal supply. New zinc concentrate production is expected to primarily come from mine life extensions and projects at financing or feasibility/permitting stages (Figure 19.3).

Accordingly, a long-term supply gap is expected to emerge which can only be reversed if prices rise to incentivise development of currently uncommitted projects (Figure 19.3).

Figure 19.3 Sources of Future Mine Supply



Wood Mackenzie, 2021

Following a 0.4 Mt deficit in 2020, the zinc concentrate market is forecasted to be in surplus in 2021 with the return to full production for those mines impacted by COVID-19. The concentrate surplus is expected to grow further in the near term with the ramp-up of new mines. However, in the long-term, a supply gap is expected to emerge which can only be reversed if prices rise to incentivise development of currently uncommitted projects (Figure 19.3).

19.1.4 Treatment Charge Outlook

The Kipushi 2022 FS assumes that zinc concentrate will be sold at industry standard terms. A long-term concentrate treatment charge of $190/dmt concentrate has been assumed.

19.2 Kipushi Zinc Concentrate

19.2.1 Concentrate Quality Considerations

For smelters and refiners, concentrate quality is an issue from both an environmental and metallurgical perspective. While not all regions of the world operate to the same environmental standards, growing pressure from international trade groups, project lenders, NGOs, and others means it is becoming increasingly difficult to place concentrates containing material levels of deleterious impurities such as iron, lead, mercury, and cadmium.

From a metallurgical perspective, smelters typically look at a feed blend to fit their metallurgical requirements.

While concentrate grades that fall outside these specifications can often be processed, smelter interest in them may be more-limited because the concentrates will either have to be subject to higher cost processing or blended with other inputs to ensure an appropriate furnace feed mix. Individual smelters may be even more restrictive on certain deleterious elements due to their own particular process technology, feed mix, and/or local regulations.

Penalty rates for impurities in zinc concentrates will vary from smelter to smelter depending on various factors including individual smelter process capabilities, existing capacity for additional inputs of a given impurity, and prevailing market conditions.

Precious metal content in concentrates can be a constraining factor as well. While not typically a metallurgical or environmental issue, the presence of high levels of precious metals may be an economic issue for certain smelters and refiners. Not all zinc smelters have precious metal recovery capability (or recoveries may be poor), gold and silver accountabilities in zinc concentrates can vary from buyer to buyer.

Based on the KICO marketing analysis there are no material quality issues foreseen with the concentrates:

- The projected zinc grade will be attractive to smelters.
- The silver and gold levels in the concentrates are projected to be low and below typical smelter payables.

- The projected germanium levels in the concentrate are higher than typical, but are, nonetheless, unlikely to be payable as very few zinc smelters actually recover germanium. While germanium may not be payable, the few smelters that do recover it may be prepared to offer a credit via somewhat lower treatment charges in recognition of the value they will derive from the germanium in the concentrates.
- Fluorine is well above typical penalty thresholds (300–500 ppm) so would likely be subject to penalties, but this is not viewed as a significant impediment; MgO levels are also slightly elevated so could also be subject to penalties; all other assays for deleterious elements are under typical penalty thresholds. Potentially concentrate with low fluorine levels could be purchased and blended to reduce the overall contained fluorine below the penalty threshold.
- Iron and lead levels are both below typical penalty thresholds.

19.2.2 Concentrate Sales Strategy and Distribution

There is currently no African smelter to which the Kipushi concentrates can be reasonably shipped. Although freight differentials will clearly come into play when determining the most suitable buyers for the Kipushi concentrates, the differentials are not deemed wide enough to strongly favour one geographic market over another. Furthermore, with the life-of-mine (LOM) annual production average of 437 kt concentrate, the Kipushi Project has the potential to be one of the largest zinc mines in the world and should look to have exposure to all the major markets.

Most, if not all traders will offer early payment for concentrates and will typically offer more competitive commercial terms (treatment charges, penalties, etc.) than smelters in exchange for delivery destination options and quotation periods. While the Kipushi concentrates are relatively clean and can likely be placed directly with most smelters. Traders are regular buyers of such products, which they can either use as a diluent for their blend(s) or for direct sale opportunities, and will frequently bid aggressively to secure supplies.

A combination of short, medium, and long-term contracts are seen as the most desirable concentrate sales offtake structure.

Based on projected annual production volumes, it would be highly unusual to contract the production to a single buyer. To diversify counterparty risk and to expose Kipushi zinc concentrates to different market regions, the output would be sold to several different buyers under staggered contract durations, avoiding multiple contracts falling due at the same time.

To manage concentrate sales in terms of contract duration and distribution a marketing strategy needs to be developed and implemented to meet the specific requirements of the Kipushi Project while taking into consideration prevailing market conditions at the time contract discussions are entered into.

As treatment terms (payable metals, annual treatment charges, escalators, etc.) can be expected to be relatively similar for all buyers of seaborne zinc concentrates, decisions regarding the ultimate distribution of the Kipushi zinc concentrates can focus on desired or preferred partnerships with specific buyers. With treatment terms relatively consistent from one buyer to the next, ocean freight rates should effectively be the only factor significantly differentiating the rates between the alternative destinations.

Although cost differentials are foreseen for deliveries of Kipushi zinc concentrates to the major market destinations, i.e. Europe and Asia, the projected differential is not viewed as significant enough to warrant a focus on one specific geographic region over the other. While consideration should be given to maximising opportunities that may be available in certain markets, (e.g. east coast South America and even North America), for strategic reasons it may be preferable for Kipushi to be active in several different zinc markets.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

The Kipushi area is of humid subtropical hot summer climate with mild, dry winters and hot humid summers. Rainfall of approximately 1,208 mm is experienced annually in the region of Lubumbashi with the wettest rainfall months occurring from November to April and the driest weather occurs from June to August. The average annual temperatures vary between 14°C and 28°C with average annual relative humidity of 66%.

The Kipushi municipality was originally developed around an existing informally planned village. At the peak of operations, it housed a mine staff of approximately 2,500 workers and their families. The current estimate of the Kipushi population is 150,000 people. As the infrastructure design is based on 20,000 people, there is tremendous pressure on infrastructure, which has not been well maintained.

Kipushi municipality is surrounded by small scale subsistence agriculture, allocated by tribal authorities. Given the population density, there is limited fertile agricultural land available for new allocation. The informal economy in and around Kipushi is driven by small, micro, and medium enterprises (SMMEs) who trade in a variety of daily necessities. Artisanal mining of aggregates and retrieving copper from old concentrate run-off also constitute a significant economic activity with an estimated number of 30,000 artisanal miners active in and around the town.

Although there is a significant environmental legacy from previous operation of the mine, Gécamines has been exonerated by the DPEM from its environmental liabilities and there is therefore no legal obligation for KICO vis-à-vis the State with regard to the environmental obligations of Gécamines, while it was the holder of the mining right from which the Exploitation Permit derived. KICO nevertheless has to comply with its obligations resulting from applicable laws and regulation concerning in particular the protection of environment and the industrial liability of the holders of mining rights, where appropriate.

Sustainability for the Kipushi Project should focus on the urban population, including continued operation of the potable water pump station, prevention of flooding and water ponding in the community for malaria control and community health initiatives including Fionet (a mobile testing and tracking platform specifically developed for controlled, rapid response to pandemics). Support and capacity building to SMMEs and to local suppliers to the mine based in Kipushi will be prioritised. There is considerable small-scale agriculture in the impact area, and the possibility of building local capacity to expand to commercial agriculture will be investigated. In addition, support to local schools in the form of bursaries, infrastructure development and collaboration with local Universities will take priority to help develop a young work force with the mine.

20.1 Previous Work

- Environmental Report on the Kipushi Zinc–Copper mine, Democratic Republic of Congo, by The Mineral Corporation, for Kipushi Resources International Limited (KRIL), 2007.
- Etude d'Impact Environnmental et Plan de Gestion Environnmental du Projet (EIA/PGEP), PER 12234, 12349 et 12350 for KICO sprl by DRC Green – EMEC, 2011.

- Environmental Management Plan (EMP) for Tailings Processing Permits PER 12234, 12349 and 12350, by Golder Associates for KICO, 2014.
- Report d'Audit Environnmental in situ Relatif a l'Obtention de l'Attestation de Liberation des Obligations Environnmentales des PER 12234, 12249, et 12250; PE 12434 de la Gécamines Cedes a KICO sprl, Republique Democratiques du Congo, Minitere du Mines, Secretariat General de Mines, Direction de Protection de L'Environnment Miniere, 2011.

In order to comply with its environmental obligations, KICO selected the environmental Design Office approved in the Democratic Republic of Congo (DRC), GREEN-EMEC, in order to carry out the Environmental Social Impact Assessment (ESIA) and the Project Environmental Social Management Plan (ESMP) relating to its PE 12434. The present ESIA/ESMP was submitted to the DPEM for approval via the Mining Registry Office on the date of the 25 October 2011.

After the instruction from the Permanent Evaluation Committee, the study was approved by DPME through its decision No. 174/DPME of the 26 November 2011, after favourable environmental opinion No.173/DPME/2011 of CPE.

The Golder 2014 EMPP on the tailings permits and the EIA by DRC Green are considered definitive for the tailings, as these have been filed with regulatory authorities.

Although subsequent Golder reports are more current and comprehensive, these have not been filed with regulatory authorities, but are the basis for industry-standard best environmental practice policies to be adopted by KICO as the baseline before advancing to the construction and production phases of the project.

A new tailings storage facility located south of the plant area will be constructed to contain approximately 2 Mt of flotation tailings. All dense media separation (DMS) tailings produced from the zinc beneficiation will be used as mine backfill.

20.2 Force Majeure Condition

The legal condition of force majeure on PE 12434 was applied mid-2011 as a result of the mine flooding, following the failure of the main underground pumping station at approximately 1,200 mL in Shaft 5. Force majeure event affecting the Exploitation Permit remains in effect at the date of this Report.

The condition of Force Majeure suspends the legal and regulatory requirements that cannot be performed as a result of the Force Majeure. Also, during the Force Majeure impacting the Exploitation Permit, KICO benefits from reduced invoices from SNEL as a result of an agreement with SNE.

Pursuant to Mining Regulations, Force Majeure is normally lifted on notification from the holder.

In accordance with Article 2 of the decision that approved the Force Majeure for the Exploitation Permit held by KICO, this notification will be done when the underground mine and its facilities will have been refurbished.

Mining Regulations finally clarify the process whereby the period of validity of the mining right impacted by Force Majeure is subsequently extended.

20.3 Environmental Audit – Removal of Environmental Obligations from KICO

As agreed in Amendment No.5 to the JV Agreement wherein 'Gécamines shall obtain from the relevant government authority, in order to release it from its environmental obligations in relation to the metallurgical and mining operations carried out before the Implementation Date, a 'declaration of release from environmental obligations' and it shall hand this over to KICO before the Implementation Date'.

Gécamines obtained this release from the DPEM on the perimeters covered by the mining rights that are currently held by KICO, including the Exploitation Permit, in August 2011 with the conclusion:

Therefore and notably pursuant to Article 405 alinea 3 of Mining Regulations, KICO is only responsible for the environmental impacts going forward, in accordance with applicable law and the update social and environmental impact assessment and management plan to be finalised and approved.

20.4 ESHIA Baseline Study

Golder Associates Africa has completed several reports on the Kipushi Project, including:

- Environmental Baseline (as at November 2011) and Liabilities Assessment.
- Environmental Management Plan (EMPP) Kipushi Tailings, February 2014.
- Assessment of Potable Water Supply infrastructure, August 2012.
- ESHIA Baseline Study, May 2015 including components of:
  - Aquatic Biology Assessment
  - Visual Baseline
  - Terrestrial Ecology
  - Radiological Baseline
  - Health Impact Assessment
  - Noise study
  - Social Risk Assessment
  - Socio-Economic Baseline
  - Geochemistry Baseline
  - Surface Water baseline
  - Stakeholder Engagement Plan
  - Groundwater Baseline
  - Air Quality baseline
  - Soil and Land-use baseline

The ESHIA Baseline study used the International Finance Corporation (IFC) guidelines as a standard, which includes the Equator Principles version 3 (EP3); with the exception that no primary health data in the Kipushi impact area were collected.

The primary impacts on the natural and social environment due to mining and related industry were considered to be:

- Air quality: Fugitive dust from historical Tailings Storage Facilities (TSFs), unsurfaced roads, air pollution from vehicle traffic, clay brick firing, veldt fires, and charcoal burning. It was noted in the 2012 report that zinc concentrate was stockpiled on site, with large amounts of mineralised dust present.
- Land use: progressive urbanisation and loss of area available for agriculture, ownership issues, lack of soil fertility (natural), caused (in part) by population influx due to economic opportunities in the mining sector.
- Surface Water: Kipushi Mine water discharge is generally within DRC regulatory discharge limits, and there is additional settling and filtering by the wetlands in TSF.
- Groundwater: contamination of groundwater by infiltration of surface water through the TSFs due to the mine dewatering.
- ARD: although the tailings have moderate ARD potential, this is generally mitigated by the neutralisation capacity of the host dolomite rocks.
- Noise: two main noise sources were identified, the Shaft 4 surface ventilation fan, and the CMSK Concentrator when operating. The CMSK plant has since seized operations and an additional ventilation fan installed.
- Radiation: although localised sources of elevated radiation were identified, the average dose rates fall within the average global dose rates.
- Biological Environment: deforestation and degradation of natural habitat resulting in loss of biodiversity, due to population influx and lack of land management.
- Socio economic environment: economic dependence on mining related business.
- Health Concerns: Malaria remains the highest mortality cause, followed by TB, and STDs (including HIV/AIDS/ARC), exacerbated by poor quality health care, although not a direct impact caused by mining, the loss of the paternal legacy of state-owned enterprises increased the concerns.
- Artisanal Miners: volatile and vulnerable group comprising some 20% of the local population as primary or supplementary means of livelihood, KICO has a good working relationship with formalised cooperatives.

20.5 KICO Internal Studies

KICO has also undertaken several studies to complement the Golder ESHIA Baseline Study, including:

- Annual survey of primary, secondary, and tertiary schools in the district, including enrolment, available capacity, and tuition fees.
- Socio-economic study of the artisanal mining population.

- SMME survey of local small businesses.
- Survey of health care facilities.
- Survey of Employee's residence locations and proximity to medical service providers.

20.6 Environmental and Social Impact Assessment

Article 463 of the Decree No. 18/024 of June 8, 2018 modifying and completing the Decree No. 038/2003 of March 26, 2003 of the Mining Regulations notably provides that the holder of a mining right is required to revise its approved Environmental and Social Impact Assessment (ESIA)and Environmental and Social Management Plan (ESMP) every five years, or when changes in the mining activities of the holder of the mining right justify an amendment of the Environment and Social Impact Assessment.

Therefore, the project developer, KICO has appointed an Environmental Design Office approved in DRC, Congo Environment and Mining Consulting (CEMIC SARL) to prepare a draft updated Project ESIA and ESMP, in accordance with the requirements set out in Annex VIII of the 2018 Mining Regulations (Decree No. 018/024 of 8 June 2018) were respected for the preparation of this updated study, the finalisation of which is ongoing.

The directive relating to its development prescribed in Annex VIII of the Mining Regulations includes eight main points mentioned as follows:

1. Awareness of the directive on the Environmental and Social Impact Assessment during the preparation of the latter.
2. Presentation of the project (project details).
3. Analysis of the environmental system affected by the project.
4. Analysis of the environmental impacts of operations.
5. Rehabilitation and mitigation measures programme.
6. Detailed budget and financial plan for the rehabilitation and mitigation programme as well as the financial guarantee for the environmental rehabilitation.
7. Consultation of the population during the preparation of Environmental and Social Impact Environmental Assessmnet and of the sustainable development plan.
8. The certification of compliance.

KICO will also have to prepare and negotiate, in a timely manner, a cahier des charges defining the social responsibility vis-à-vis local communities impacted by the activities of the Project in accordance with applicable laws and regulations.

20.7 Tailings Management and Disposal

20.7.1 Design Criteria

The TSF is to be designed to contain the tailings stream based on a run-of-mine (ROM) of 181 ktpa and a LOM of 13.8 years. As per the production schedule provided by Wood, the expected total production of dry tailings are:

- 2.6 Mt of Zinc tailings, relating to
- 1.3 Mm^3 of Zinc Tailings.

The expected slurry characteristics of the Kipushi tailings are listed in Table 20.1.

Table 20.1 Tailings Slurry Characteristics

Parameter	Acidic Tailings	Source
Solids Specific Gravity (SG)	4.036	STL (Test work)
Final Void Ratio	1.11	BMPCE (Test work)
Placed Dry Density (t/m^3)	1.994	BMPCE (Test work)
Placement	Hydraulic	METC
% Solids to Water by Mass	55.0	METC
Design Deposition Rate (average dry t/month)	15,675	METC
Volume of Slurry (m^3/month)	28,500	Epoch (Calculated)

20.7.1.1 Waste Classification of Tailings

Waste classification of Kipushi tails:

- The tailings sample classified as potential acid generating based on 17% sulfide (S) and Neutralisation potential ratio <1. Kinetic test work confirmed that the acidity would not be realised (week 0–week 17 pH = 5.95–6.99) due to neutralising surplus.
- The metal leaching classification was originally classified as high risk due to Toxicity Characteristics Leaching Procedure (TCLP), concentration of Lead (Pb) = 48 mg/L exceeding the high risk criteria value of 5 mg/L, with Zinc (Zn) = 50 mg/L exceeding the low risk value of 1 mg/L which classified the sample as a leachable mining waste based on TCLP tests (acidic conditions).
- Kinetic results indicate dissolved Pb and Zn concentrations of 0.01–0.21 mg/L and 0.3–6.4 mg/L respectively, implying that the tailings is not classified as low or high risk but rather as 'leachable mining waste' which refers to 'slightly hazardous' according to the DRC regulations.
- The tailings material is reclassified based on kinetic tests as not potential acid generating and leachable mining waste (Level-A permeability measure applicable depending in the underlying soil permeability).

20.7.1.2 Site Selection

Approximately 2.6 Mt of flotation tailings will be stored in a new TSF. Several sites provisionally identified for locating the TSF are shown in Figure 20.1.

Figure 20.1 Potential Tailings Dam Locations – Site 4 Selected for the Study

Golder, 2019

The site selection for the Kipushi TSF was based on the following criteria:

- Tailings deposition rates.
- Suitable topography for storing the required capacity.
- The nature of the material to be deposited and its short and long-term environmental impacts.
- The nature and sensitivity of the surrounding environment, i.e. the receiving environment.
- Avoiding the following:
  - Planned and Existing mine infrastructure.
  - The Zambian-DRC border.
  - Potential underground mining within a depth of 200 m to Natural Ground Level (NGL).
  - Geological anomalies such as active faults and shear zones.

- The underlying soil properties, i.e. the need for liners and compaction.
- Possible flood lines.
- Potential reserves.
- A risk analysis was performed, which took cognisance of the surrounding environment, people, and nearby infrastructure.
- High level costs associated with the construction and operation of the TSF.

Site 4 was selected as the preferred location for the following reasons:

- Financial: The facility is positioned relatively close to the process plant, with achievable pumping heads required for the tailings delivery line. Secondly, as a result of the topography, this footprint provides the full impoundment facility with the second smallest required fill volume (impoundment wall) and the smallest area required to be lined.
- Geological: Site 4 is positioned on an area with no known geological anomalies within its footprint, which contributes to its favourable rating in this category.
- Visual impact: The final height of the facility is relatively low and positioned away from houses.
- Potential for expansion: The site is positioned in an area providing the possibility for significant expansion.

During detailed engineering and execution, the KICO team will also be considering Site 1.

20.7.1.3 Construction Materials

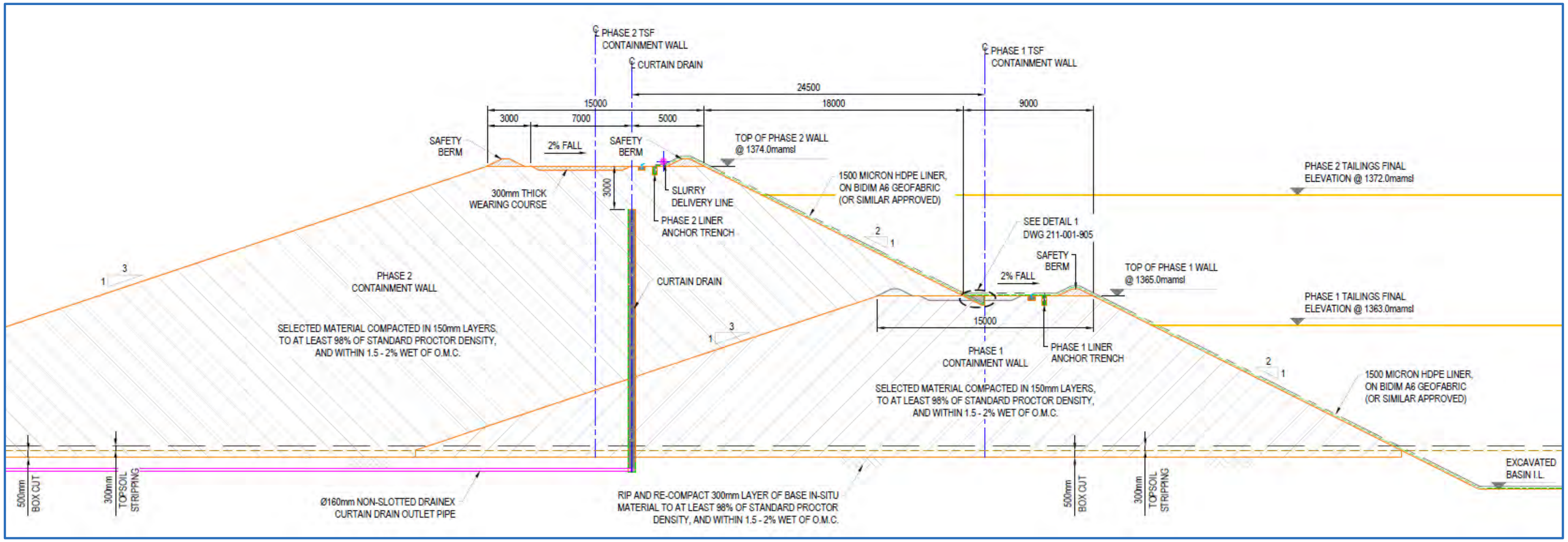
The construction material volumes from the basin of the TSF together with the estimated volumes from the proposed borrow areas will provide a sufficient amount of material for the construction of Phase 1 of TSF Site 4. As some of this material will be used for terracing in other areas of the mine, further investigation will be required in the area to identify additional borrow areas to provide enough engineered fill material for the construction of Phase 2.

20.7.1.4 Site Development Strategy

It is envisaged that the TSF will be constructed as a full containment type facility with material for the containment walls sourced from the basin of the TSF. The balance of material from the cut-to-fill construction operation to be sourced from approved borrow areas containing the specified material.

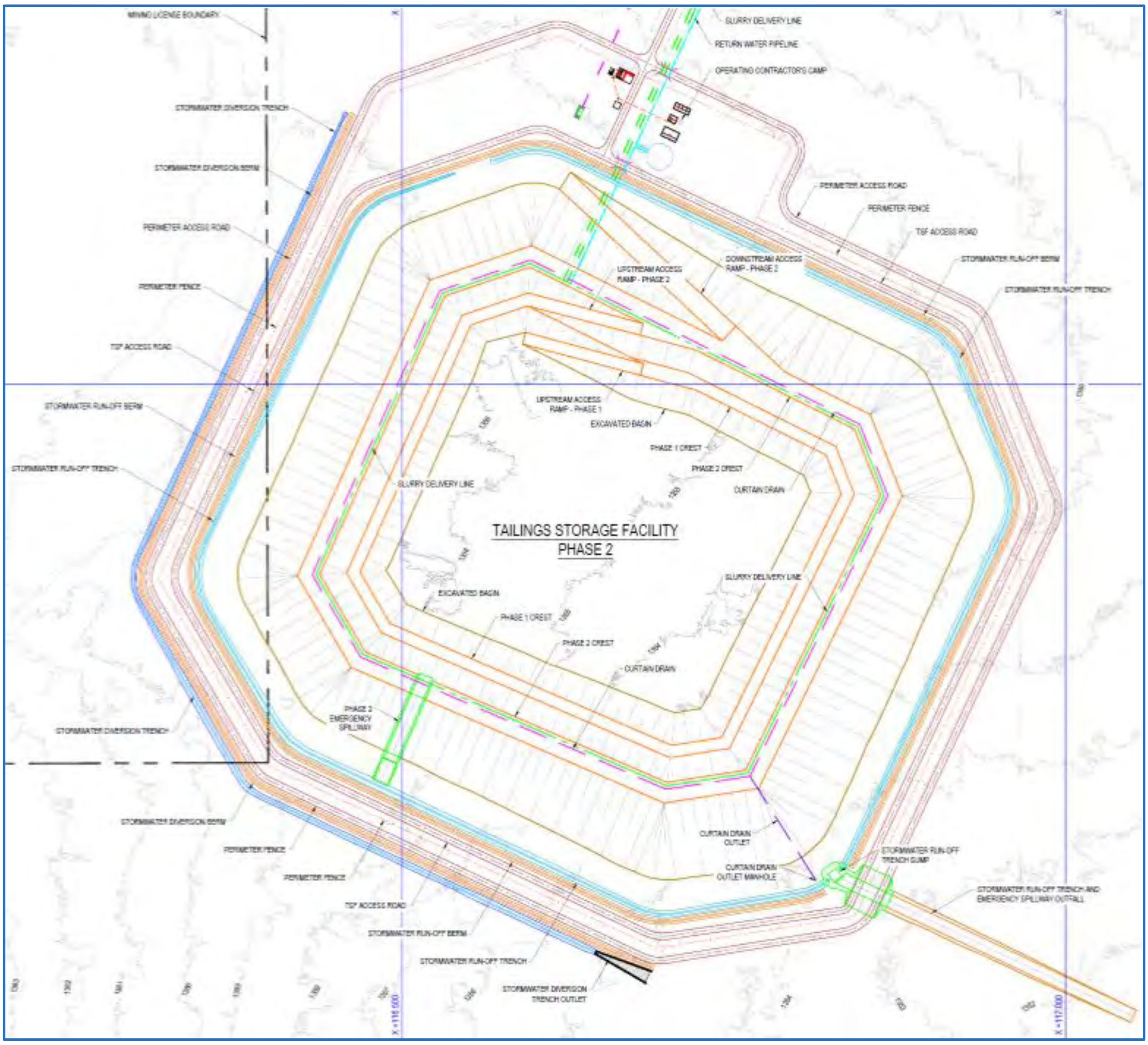
The TSF will be constructed in two phases with the containment walls being developed in a downstream construction method as illustrated in Figure 20.2. Phasing the construction allows for early tailings storage volume without requiring the entire developed facility, while construction of a successive phase may continue, if the timeline requires it, and defers the CAPEX throughout the LOM. The general arrangement of the proposed TSF and its associated infrastructure is illustrated in Figure 20.3.

Figure 20.2 TSF Downstream Phased Containment Wall Development



Golder, 2019

Figure 20.3 General Arrangement of the TSF and Associated Infrastructure



Golder, 2019

The TSF will comprise the following:

- Phased compacted earth-fill full containment wall with material sourced from the TSF basin or approved borrow areas.
- Single liner system comprising of:
  - Base preparation of the in-situ material which will be ripped to depth of 300 mm and re-compacted to 98% of the Standard Proctor density.
  - A6 Bidim geotextile (or similar specified) secured in a liner anchor trench.
  - A 1,500 micron HDPE (High-Density Polyethylene) geomembrane across the basin and side slopes secured in a liner anchor trench.

- Phased curtain drain constructed in the containment walls of Phase 1 and 2, comprising a 160 mm slotted HDPE Drainex pipe within a coarse stone matrix overlain by an intermediate stone, overlain by a filter sand and protected by an A6 geotextile (or similar specified) on the downstream face with the upstream face exposed to the compacted impoundment wall material.
- Curtain drain outlet comprising of a 160 mm non-slotted HDPE Drainex pipe within a coarse stone matrix wrapped in an A6 Bidim geotextile with a tie-in to a pre-cast concrete collection manhole.
- Access ramp from the downstream side of each phase leading onto the respective crests.
- Pool access ramps into the basin of the facility to access the point of water extraction. Each phase will have a separate access ramp as per the phasing implemented.
- Temporary emergency spillway on Phase 1 tying into the storm water run-off trench.
- Final emergency spillway on Phase 2 also tying into the storm water run-off trench.
- A storm water run-off trench, bund wall and emergency spillway outfall structure.
- A perimeter access road (by others).
- Perimeter fence with associated vehicle and pedestrian access points (by others).

Supernatant water is to be decanted from the facility pool and either returned to the process plant or released into the environment by means of floating barge and pump system designed, supplied, installed, and operated by others.

20.8 Water Management

As part of the FS, Golder Associates updated the Surface Water Impact Assessment for the Kipushi Project. This included a baseline and impact assessment incorporating:

- Review of existing surface water monitoring programme to assess whether the main watercourses that are being (and could be) affected by the planned activities are being adequately monitored. Recommendations for additional sampling sites were made where necessary.
- The surface water data was analysed to produce statistics that characterised the flow, where possible, and water quality profiles of the potentially impacted streams. This data was collated into a database that will track the monthly changes in water quality and flow throughout the monitoring period.
- The available climate data was collated and analysed.
- Historical rainfall data from the site was compared to data obtained from other weather stations in the vicinity if available and long-term rainfall patterns were established for use in the hydrological studies.

As part of the FS, Golder Associates Africa updated the Surface Water Impact Assessment for the Kipushi Project. This included a baseline and impact assessment. As part of the impact assessment, potential surface water impacts were identified on, and external to the project area. The identified impacts were assessed and mitigation strategies formulated, taking into consideration the TSF design, dewatering, and future proposed surface water infrastructure.

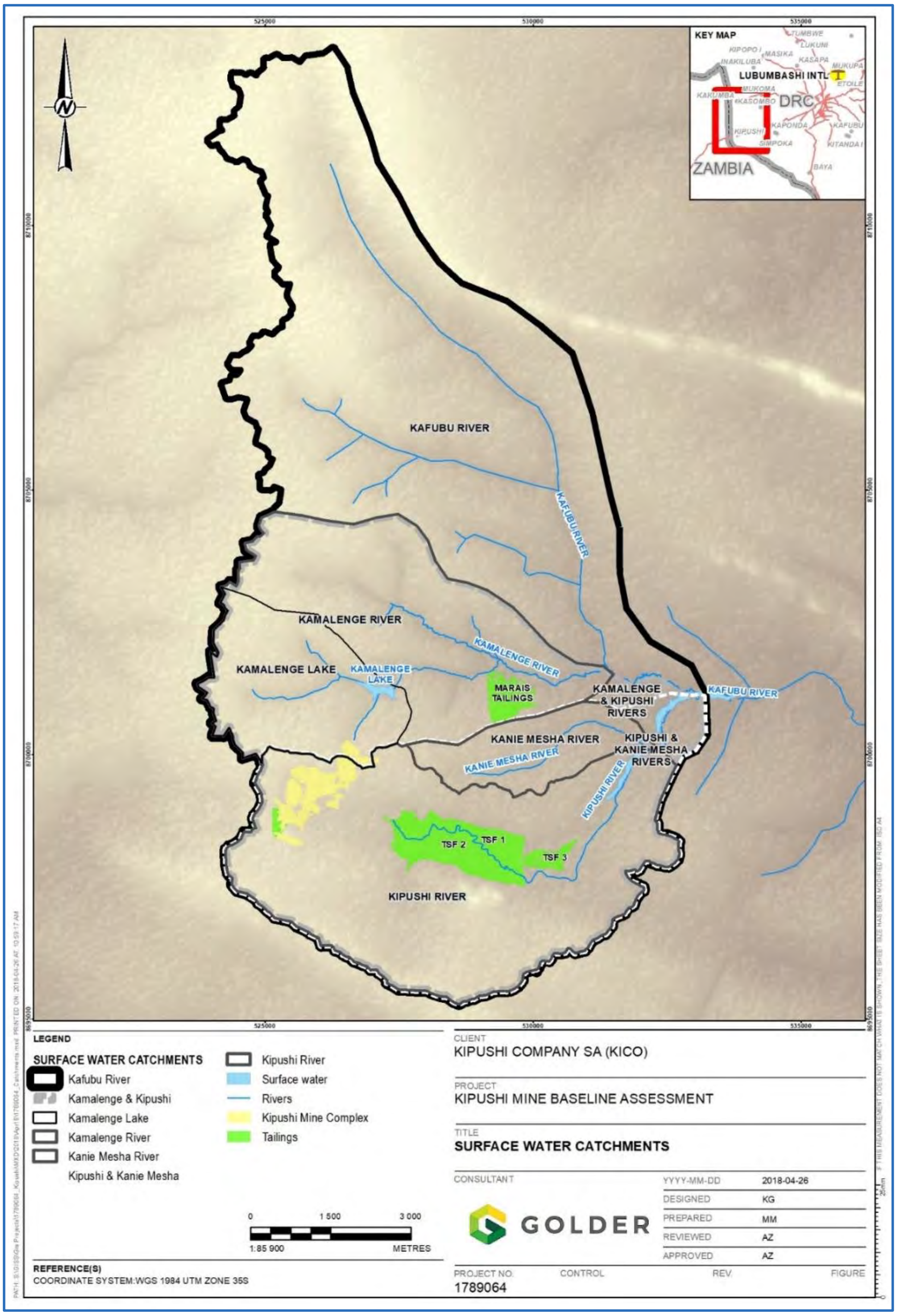
The assessment included:

- Potential impacts on surface water quality due to spills or discharges from containment structures.
- Potential impacts on surface water quantity as the result of dewatering and reduced catchment areas.
- Potential effects on downstream users.

20.8.1 Regional Surface Water Resources

The mine is located in the upper reaches of the Kipushi catchment with the existing mine tailing storage facilities located in the middle reaches of the Kipushi River. The Kanyameshi River joins the Kipushi River from the north about 3 km downstream of the TSF. The Kipushi River flows east for another 1 km before it joins the Kafubu River. The Kamalenge River flows in an easterly direction to the north of the Kipushi River catchment. The Kamalenge River is also a tributary of the Kafubu River. The Kamalenge Lake is located in the upper reaches of the Kamalenge River (also referred to as Lac Kipushi). A small area of the mine is located in the Kamalenge River catchment with the run-off draining to the Kamalenge Lake. The Kafubu River drains in a southerly direction and turns to flow in an easterly direction at the confluence of the Kafubu and Kipushi Rivers. There are extensive wetlands in the lower reaches of the Kipushi River and in the Kafubu River downstream of the Kafubu and Kipushi River confluence. The Kafubu River flows north-east towards Lubumbashi. Water is abstracted from the river to supply Lubumbashi and is used for irrigation. Figure 20.4 details the catchment areas of the rivers.

Figure 20.4 Location and Extent of the Surface Water Catchments in the Vicinity of Kipushi Mine



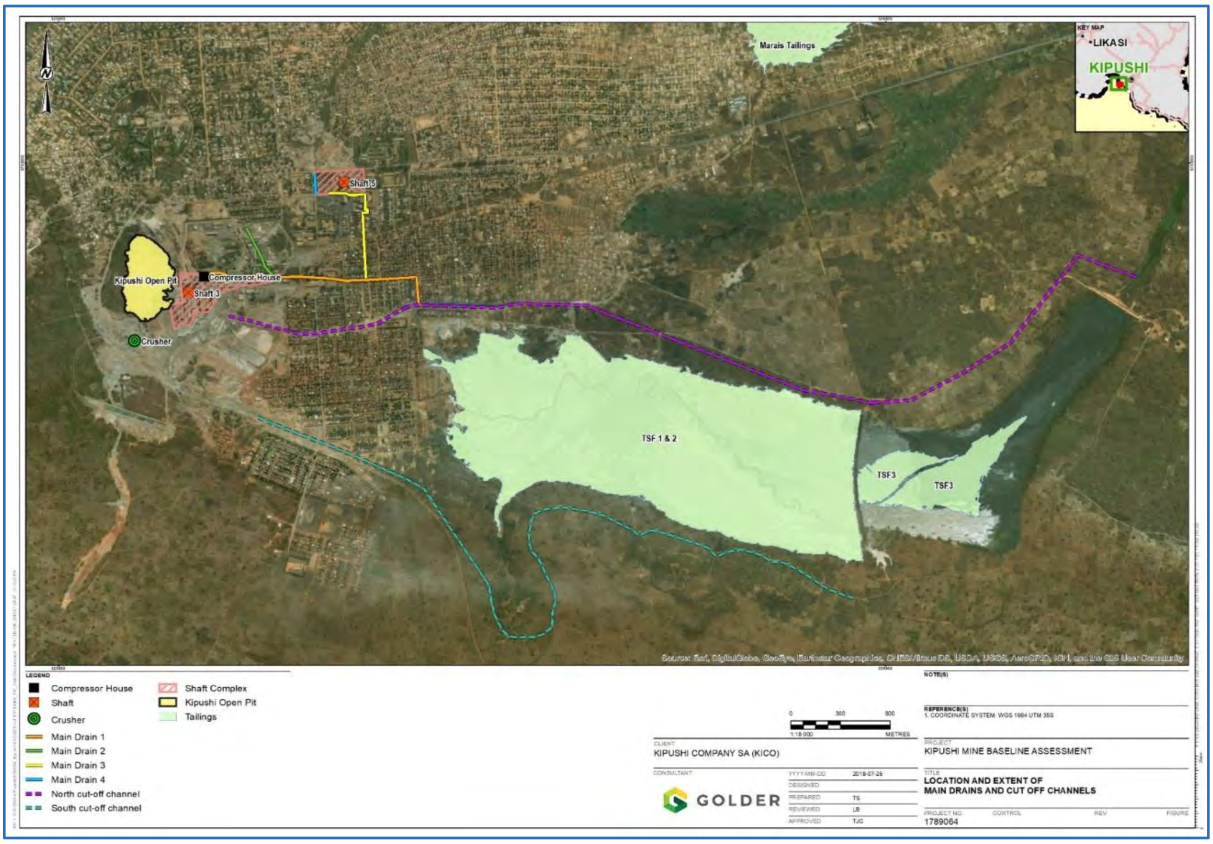
Golder, 2018

20.8.2 Mine Stormwater Management

The layout of the mine stormwater management drains are shown in Figure 20.5. Historically the stormwater run-off, and water pumped from underground, is conveyed in channels to discharge into the Kipushi River to the east of the mine complex. The drain from Shaft 5 was used to convey tailings from the CMSK concentrator for deposition on the TSF. However, the CMSK has now ceased operations. The proposed development consists of a new TSF, plant, stockpile, and waste rock storage facility. Stormwater run-off from the new infrastructure will report to the existing stormwater drainage system.

There are four main stormwater drainage channels on surface. Drain locations are shown in Figure 20.5.

Figure 20.5 Location and Extent of Existing Main Drains and Cut-off Channels



Golder, 2018

20.8.3 FS Mine Water Circuit

Metallurgical testwork conducted indicates that the raw water recovered from mine dewatering is not appropriate for use in the flotation circuit. Therefore, potable water is utilised as process water make-up. Provision is made, however, for make-up using raw water if required. Flotation tailings will be deposited in a new TSF, located south of the process plant.

In the proposed scheme (Figure 20.6), the return water from the TSF is first neutralised with lime ($Ca(OH)_2$) and blended with the excess underground water before being discharged into the Kipushi River, via the north cut-off channel.

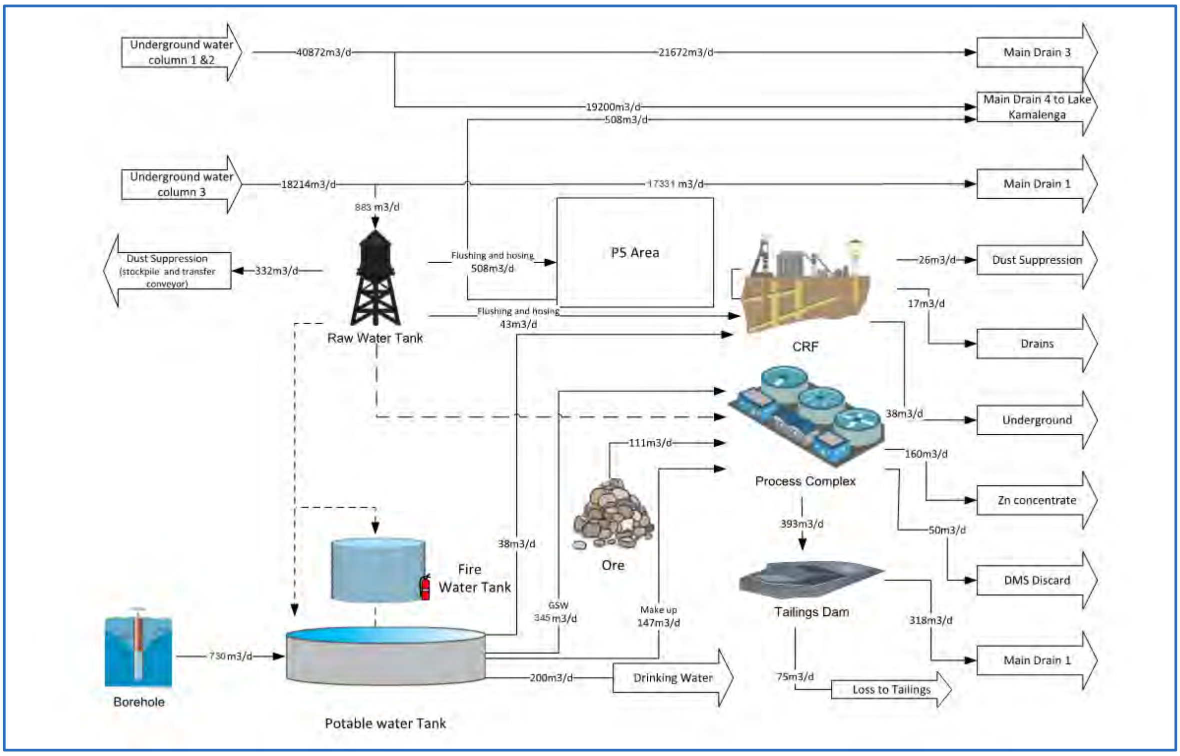
A neutralisation plant has been included in the FS, on the basis that the plant metallurgical simulations undertaken, suggest that the pyrite to dolomite content of the tails is such, that the TSF return water is likely to be acidic, and that even after blending with underground water prior to discharge, the water released to the environment would fall outside the DRC prescribed pH discharge limits.

A system of clean water channels have been designed to cut-off the clean run-off upstream of the TSF (refer to Figure 20.6). The clean water is then returned to the environment.

Water supply for the Kipushi area is obtained from a well field located approximately 1.0 km south-east of the town and south of the tailings dam. The well field was designed to have 10 large diameter boreholes drilled into the Kakontwe Dolomite and Limestone aquifer. Six of these boreholes were equipped with pumping equipment and the other four were left unequipped to be standby wells. The pumps installed are of the vertical spindle type where the pump is at the bottom of the borehole and is driven by a shaft connected to an electrical motor on surface. Water is delivered to a Central Sump.

Potable water is received from the local municipal supply, stored in the new potable water tank, and distributed to various users.

Figure 20.6 Schematic of the Kipushi Reticulation System



Golder, 2019

20.9 Geochemistry Risk Assessment

The ARD and ML potential of the mined materials and tailings material associated with planned mining activities is a function of the host geology, the Acid Base Potential of the geological materials, interconnectivity to other mine workings and mine water management measures. These influences on the mine water generated at during the operational and closure phase was assessed using available data.

The ARD and ML potential of the ore and waste rock associated with the proposed underground mining activities were assessed using available water quality monitoring data, site geology data, and geochemical data obtained from analogue sites in the Lufilian arc (south eastern) of the Central African Copperbelt.

The following mining activities will result in ARD and ML processes:

- Refurbishment, replacement and/or establishment of the following underground functions:
  - Main haulage of mined material (ore and waste rock), notably on the 1,270mL.
  - Underground mine development – decline construction, shafts, and ventilation shafts.
- Establishment of backfill passes, transfer systems, and backfill infrastructure, including cemented fills. This includes the disposal of waste rock as backfill material (waste rock and tailings produced).
- Mining of remnant ore bodies.
- Tailings disposal from the processing of the Big Zinc ore. Tailings material solid and supernatant process water samples not available at the time or reporting. The impact assessment has been done qualitatively based on the understanding the new TSF will be lined (HDPE double liner system).

The following mining activities could also lead to ARD and ML impact; however, are excluded from this assessment for the following reasons:

- Backfill material: The blending and volumes of backfill material is unknown at the time or reporting.

The following conclusion are made from the geochemical characterisation for KICO DMS and ROM material:

- The ROM material classified acid generating, whereas the DMS discard classified as non-acid generation based on DRC Mining Regulations. No kinetic test work have been conducted on the ROM and DMS discard material.
- The Metal leaching risk from the DMS discard and ROM have been assessed by the TCLP test work according to DRC Mining Regs, and Fe, Zn, and Pb concentrations exceeded the low risk criteria. Lead (Pb) concentration exceeded high risk criteria for the ROM sample classifying the ROM as High Risk. The DMS discard classifies as 'Leachable Mining Waste' implying Level A impermeability measures are required. Level B impermeability measures (HDPE liners) may be required for the ROM stockpiles since the materials classified as a high risk mine waste.

- In terms of Level A impermeability measures, the storage and management of a mine waste is permitted on unaltered soil of at least 3 m thickness with a hydraulic conductivity ≤1 x 10–6 cm/s (according to DRC Regulations 2003, Annexure IX), and may be permitted on substrate with a hydraulic conductivity of between 1 x 10–4 cm/s and 1 x 10–6 cm/s for a soil at least 3 m in thickness or a rocky base (without secondary porosity or fractures), if it is demonstrated through modelling that the in situ hydrogeological conditions limits significant degradation of the underground water quality.
- For additional impermeability measures implemented (e.g. ground improvements such as rip and re-compaction or bentonite addition to lower the in-situ permeability), the groundwater modelling study is required to demonstrate the efficiency of the proposed measures; in avoiding degradation of the underground water quality.
- In term of level B impermeability measures for the ROM stockpiles, single or double-layer geomembrane engineering designs may be required depending on the underlying soils permeability for a 6 m layer or 3–6 m layer at ≤1 x 10–6 cm/s respectively (Annexure IX, DRC Regulation 2003).

The following recommendations are made from the geochemical characterisations of the ROM and DMS discard:

- The study classified average LOM ROM as PAG. Additional ore material samples should be collected and characterised as part of geochemical sampling campaign to confirm the acid generating classification of the ROM materials and variance across the Big Zinc deposit.
- Regular monitoring the actual seepage and run-off qualities (and volumes) should be monitored from DMS discard dumps and ROM stockpiles, during wet seasons to confirm the field pH and inorganic chemical concentrations.
- Kinetic test work should be conducted for ROM and DMS discard material to confirm the metal leaching concentrations in the medium to long term, and NP depletion rates.

The following general recommendations are made for the KICO operations:

- Continuation of ARD monitoring and static geochemical characterisation for waste rock (by mining unit) and tailings material. The monitoring programme should include the following activities and materials:
  - Water quality of stormwater storage facilities.
  - TSF pool water quality.
  - TSF seepage water volume and quality from toe seep and/or piezometers installed around the beach sections.
  - Water quality accumulating in or being pumped from the underground mine levels.
  - General surface and groundwater monitoring.
  - Ad hoc monitoring of toe seepage from the Waste Rock Dumps.
- Despite the Non-PAG characteristic of the DMS waste rock it is recommended that consideration be given to conduct field kinetic testing using barrels or field plots of DMS waste rock to develop site specific drainage characteristics of the waste rock material.

The geochemical impact assessment indicates that ARD and ML risks are associated with the proposed mining activities at the Big Zinc Underground Mine. These ARD and ML risks could lead to mine water qualities that could exceed DRC effluent guidelines and will require mitigation.

20.10 Air Quality Impact Assessment

An air quality management plan was developed as part of the FS, providing guidance for the implementation of a framework of recommendations to preserve the physical environment in terms of air quality within the Kipushi Mine footprint. Adherence to DRC mining regulations is a legal requirement, recommended targets are IFC Performance Standards (IFC, 2012); and IFC EHS Guidelines (IFC, 2007).

The key objective of the air quality management plan is to avoid, prevent, or reduce harmful impacts on human health, and the environment as a result of Kipushi operations. The following processes were identified as significant sources of emissions and should be targeted:

- Materials transfer,
- Crushing and Screening,
- Tailings and stockpiles,
- Emissions as a result from vehicle activity,
- Backup power generation, and
- Wind Erosion.

Mitigation measures for particulates include:

- Wet suppression and wet misting during materials handling activities.
- Covering of conveyors.
- Wind speed reduction through sheltering or wind breaks for open exposed areas prone to wind erosion i.e. ROM stockpiles etc. (where possible).
- Covering or keeping stockpile heights as low as practicable to reduce their exposure to wind erosion and thus dust generation.
- Progressive rehabilitation and re-vegetation.
- Reduction in unnecessary traffic volumes.
- Wet suppression on all unpaved roads with water or a suitable dust palliative to achieve 50% control efficiency or better (note: water alone will only achieve a 75% control efficiency).
- Rigorous speed control and the institution of traffic calming measures.

Mitigation measures for trace gasses include:

- Maintain and service all mining vehicles, backup power generation, and other equipment regularly to ensure that emissions are kept to a minimum.

- Where possible, use low sulfur fuels to reduce SO_2 emissions.
- Ensure no burning of waste materials on site.

Ambient air quality monitoring have been conducted in the Kipushi Project area. This air quality monitoring programme should continue and be expanded to cover the new TSF.

20.11 Biodiversity Impact Assessment

Potential negative impacts on biodiversity that may result from the proposed project are listed in Table 20.2.

Table 20.2 Identified Impacts and Project Phases that may Manifest

Impact	Phase		
	Construction	Operational	Closure
Loss or disturbance of natural and semi-natural habitat caused by vegetation clearing and earth works.	X		
Establishment and spread of alien invasive plant species.	X	X	X
Increased sediment runoff into rivers.	X	X	X
Water quality deterioration (Contamination of surface water entering rivers).	X	X	X

The management measures presented in Table 20.3 have been developed to mitigation potential impacts of biodiversity associated from the proposed project.

Table 20.3 Recommended Mitigation Measures

Impact	Mitigation Measure
Loss or disturbance of natural habitat caused by vegetation clearing and earth works	• Vegetation clearing should be restricted to proposed infrastructure footprints only, with minimal clearing permitted outside of these areas. • Areas to be cleared should be clearly demarcated to prevent unnecessary clearing outside of these sites. • Where practical all infrastructures should be sited to avoid disturbing active termite hills; and • The provisions of the Golder developed rehabilitation programme should be followed in all disturbed areas post mining.

Impact	Mitigation Measure
Establishment and spread of alien invasive plant species	• An alien invasive species control programme must be developed and implemented around mine operational sites. The programme will target areas that have been disturbed by mining and where the establishment of alien invasive species is likely; and • The programme will include periodic follow-up treatments informed by regular monitoring.
Increased sediment runoff into rivers	• Suitable surface water infrastructure should be constructed to ensure that all storm water on-site is appropriately channelled around construction sites. • Implement perimeter sediment controls, such as the installation of sediment fences along downslope verges of the construction site. • Discharge stormwater from the construction site (dirty water) into adjacent grassland rather than directly into wetland habitat. Discharged flows must be slow and diffuse. • Regular inspection and maintenance of sediment controls; and • Rehabilitation (incl. site stabilisation and revegetation) must be implemented in all disturbed areas post construction and mining to minimise potential erosion.
Water quality deterioration (Contamination of surface water entering rivers)	• Ensure that no equipment is washed in the streams and wetlands of the area, and if washing facilities are provided, that these are placed no closer than 200 m from a wetland or water course. • Potential contaminants used and stored on site should be stored and prepared on bunded surfaces to contain spills and leaks. • A management and mitigation plan for spillages or possible overflow events should be developed or updated to include the proposed new activities; and • Water quality monitoring should be undertaken of rivers and streams, upstream, and downstream of proposed mining operational areas to measure potential water pollution that may affect downstream aquatic habitats, as per the existing Golder developed monitoring programme.

20.12 Soil, Land Capability and Land Use Baseline and Impact Assessment

Over the entire study area, the predominant land use is rain-fed agriculture, covering 60–65% of the total area. The four types of land use in the Plateau landscape are:

- Rainfed agriculture: 44.3%,
- Fallow (contaminated degraded land): 36.6%,
- Degraded forest: 5.3%, and
- Parcelling out: 13.8%.

The three types of land use in the mountainous landscape are:

- Rainfed agriculture: 78.2%,
- Career for rubble: 12%, and
- Fallow (contaminated degraded land): 9.8%.

The majority of proposed surface infrastructure at the Kipushi Mine site will be founded on previously disturbed footprints is likely to comprise of Anthrosols. The footprints of the TSF Site 02 and Site 03 are located on land that is undisturbed and suspected to be used for agricultural practices. The baseline status of the areas where the proposed infrastructure will be developed is not fully understood since it did not form part of the 2012 Baseline Assessment. However, it is likely that the area have been impacted by historical operations.

Given that insufficient baselines soils and land capability data exist, the magnitude and assessment of impacts likely to arise from the development of the TSFs could not be established. However, the anticipated impacts include changes to current land use, soil quality degradation, soil contamination and erosion. The current land use of the area within and around the TSF is likely to be rain-fed agriculture.

It is recommended that a soils specialist study be conducted to gain an understanding of the baseline soil and land capability of the area within and around the TSF sites. The data from the baseline study will be used to assess and evaluate the impacts arising from the development of the TSFs and associated infrastructure and to finalise the TSF site selected. An infill baseline contamination status assessment which will be used in conjunction to the 2012 and 2015 studies is required to establish and update the contamination status of the proposed areas.

20.13 Visual Impact Assessment

The Kipushi landscape is highly modified and developed, with a typical range of residential and commercial buildings, as well as various mine infrastructure, dominating its visual character. Beyond the immediate built environment, land comprises a mosaic of cultivated land, secondary grassland and shrub and localised pockets of woodland. The visual resource value of the study area varies from moderate to low.

Of the proposed project activities, the development of the TSF (depending on option selected) could have a noteworthy negative impact on the visual environment over the long-term. Successful rehabilitation of this facility, which must include the establishment of a stable vegetation cover, will however reduce the envisaged visual impacts post-decommissioning.

Secondary impacts, such as dust emission and lighting at night, will also manifest as visual disturbances from project initiation. However, these can be mitigated with the effective implementation of recommended management measures.

20.14 Noise and Vibration Impact Assessment

The impact assessment revealed that the establishment of additional mining activities and the construction of a TSF to the south of the mine will be moderate after mitigation measures. The impact assessment revealed that the establishment of additional mining activities as well as the construction of storage activities, on the southern side of the mine, will have a moderate impact after the implementation of mitigation measures. This will ensure that the natural environment will not be compromised where the health and well-being of the residents of the sensitive receptors will be affected.

The overall risk of the proposed mine establishment at Shaft 1, 2, 3, 4, 5, TSF, and road upgrade will be major and will become moderate with the implementation of noise mitigatory measures.

The following recommendations are provided:

- Environmental noise and ground vibration monitoring at the two shafts footprint areas and at the abutting noise receptors to be carried out.
- The environmental noise and ground vibration monitoring programmes should be structured around comparison to and in compliance with local (DRC) standards and international guidelines (IFC) as observed levels exceed assessment levels and may be impacting negatively on human health.
- KZP's process contributions and the level of impact of the noise require quantification by means of the implementation of a management plan to identify environmental noise and ground vibration problems on a pro-active manner.
- A formal noise and vibration management programme (NVMP) for the KZP mine should be developed, implemented, and included into the mines EMP.

Management intervention will require that a noise and vibration baseline study be undertaken at the major noise sources and along the boundaries of the shaft footprint areas, along the route to the mine and at the noise sensitive areas during the day and night-time periods. The noise and vibration survey will be required to implement engineering mitigation measures on a pro-active manner.

The upgrade of the existing mine shaft footprint areas, the upgrade of the existing road and the establishment of a new TSF south of the mining area will be in line with the International Health and Safety requirements as well as the DRC Mining Code requirements provided that all the mitigatory measures and recommendations are in place and maintained.

20.15 Radiation Management Plan

The radionuclides responsible for the radiological risks associated with the Kipushi Mine operations are uranium 238 (238U), uranium 235 (235U) and thorium 232 (232Th). Not all radionuclides in these series are important because only a selection of these contributes significantly to the total dose received by a person. This selection is:

- Long-lived alpha (α) emitters: 238U, 234U, 230Th, 226Ra, 210Po, 231Pa, 227Ac, 223Ra, 232Th, 228Th, 224Ra.
- Beta (β) emitters: 210Pb, 228Ra.
- 222Rn and 220Rn (and their short-lived daughters).
- Gamma (γ) emitters: 214Pb, 214Bi, 228Ac, 208Tl.

The sources of these radionuclides are generated fugitive dust emissions, such as road construction, vehicle entrainment and blasting, and disturbed and undisturbed natural soils and minerals where construction would take place.

A Radiation Management Plan (RMP) is required in terms of Article 404 (Decree N° 18/024 of 8 June 2018 Amending and Completing the Decree N° 038/2003 of 26 March 2003 before Mining Regulation) on mining regulation on the protection against ionising radiation hazards and on the physical protection of nuclear materials and facilities. Guidance on the structure and contents of the RMP is provided in [14]. A RMP for the Kipushi Project was developed as part of the FS.

20.16 Mine Closure Analysis

For the FS, the closure planning and associated costs were updated by Golder for Kipushi, using the latest available information and current closure-related requirements. Field work was undertaken for several specialist disciplines to fill existing information gaps, specifically in the case of the surface water, groundwater, and geochemistry as well as climate and air quality, which informed the current closure plan update.

The following specific objectives guided the compilation of this rehabilitation and closure plan:

- Develop a closure-focussed understanding of the latest baseline and other available background information, to establish specific closure requirements.
- Formulate an appropriate closure scenario and identify suitable post-mining (next) land use/s of the rehabilitated mine site, based on the likely prevailing conditions at the planned LOM.
- Identify screening-level closure-related risks to determine possible unwanted occurrences and associated resultant impacts, that may negatively impact the mine site during and after closure.
- Develop a corresponding closure vision and objectives based on the above and identify specific operational rehabilitation and closure measures towards achieving these.
- Identify monitoring, maintenance and aftercare requirements and associated measures aligned with the above.

- Calculate the resultant mine closure costs based on the previous closure costs update.

Mine closure planning should be viewed as an iterative process, whereby the overall closure approach and specific measures are continuously updated and progressively refined, as conditions change over time and new information becomes available. Solutions should continually be refined via a cyclical process of survey, analysis, design, and evaluation based on implementation.

A closure scenario was developed for the Kipushi Mine using a snapshot of three different time periods as explained below and reflected in Table 20.4.

- A description of the site on the last day of planned LOM operations, which forms the starting point or baseline against which decommissioning, and closure activities will follow. The importance of executing the various preparatory operational requirements in anticipation of site closure to follow becomes evident, as failure to do so will have a knock-on effect during decommissioning and closure that can cause unwanted delays and complications.
- Key activities and actions that will take place and measures that will be implemented during the decommissioning and closure period.
- A description of the anticipated post closure character and nature of the rehabilitated site, as well as remaining activities to be implemented to progress the site to a stable and self-sustaining state for eventual site relinquishment.

The preliminary rehabilitation and closure plan define the context and sets the tone for rehabilitation and closure planning to be initiated and advanced by Kipushi during operations towards a final closure plan for implementation at cessation of operations.

Table 20.4 Kipushi – Project Closure Scenario

On Last Day of Operations	During Decommissioning and Closure	Post Closure
Plant and Surface Infrastructural Areas		
• All surface infrastructure becoming redundant during operations (if applicable) will have been appropriately dismantled and demolished, and the remaining disturbed footprint areas would have been rehabilitated to match the requirements of the defined next land use. • Kipushi will have limited or no stockpiles left and the plant will have been run down and be available for demolition and dismantling. • Product export by rail would have ceased and the railway siding will become defunct. • Contractors for surface infrastructure demolition and rehabilitation would have been appointed, based on a scope aligned to the final closure plan. • The inventory of infrastructure for transfer to the local community and government (if any) would have been finalised and appropriate handover agreements would already be in place. • An appropriate and comprehensive demolition and waste disposal solution would have been identified and required permit and other requirements addressed. • A dedicated decontamination and waste screening area would have been created.	• Demolition of all infrastructure not earmarked for reuse will take place, and the resulting footprint areas will be rehabilitated. Infrastructure to be demolished and rehabilitated broadly includes the plant, all on-site buildings, stockpiles, conveyors, Grindrod siding, P2 and P5 shafts and related infrastructure. • Substations, transformers, switchyards, powerlines, and roads will be handed over to government for management. • Decontaminated steel and related material from plant demolition having salvage value will remain on-site for sale. • Demolition waste will be decontaminated within the dedicated decontamination area and disposed within the onsite waste cell, as per the final waste disposal solution. • Any contaminated soils found within the plant area will be excavated, remediated on site if feasible, or otherwise dealt with as per the final waste disposal solution.	• Monitoring will take place to confirm success of closure measures implemented at the site, until performance objectives and abandonment criteria are met. • Care and maintenance will be implemented, based on monitoring results. • Site relinquishment will take place upon receipt of a closure certificate.

<table>
<tr><th>On Last Day of Operations</th><th>During Decommissioning and Closure</th><th>Post Closure</th></tr>
<tr><td colspan="3">Tailings Storage Facility (TSF)</td></tr>
<tr><td>Topsoil would have been stripped from the planned new TSF footprint and other infrastructure development areas and managed as stipulated by a dedicated soil stripping and management plan and stockpiled appropriately to retain the soil properties.
The above soil stripping plan will indicate to what depth the soils should be stripped, as determined by the pedology of the soil profile.
The new TSF will be at full to capacity and tailings deposition will have ended.
The detailed post mining landform design would have been completed, the side slopes will not be reshaped and are ready to receive the specified cover material and only the top surface requires reconfiguring, capping and revegetation. If feasible, any initial concurrent revegetation of the side slope will also have been implemented.</td><td>Topsoil would have been stripped from the planned new TSF footprint and other infrastructure development areas and managed as stipulated by a dedicated soil stripping and management plan and stockpiled appropriately to retain the soil properties.
The above soil stripping plan will indicate to what depth the soils should be stripped, as determined by the pedology of the soil profile.
The new TSF will be at full to capacity and tailings deposition will have ended.
The detailed post mining landform design would have been completed, the side slopes will not be reshaped and are ready to receive the specified cover material and only the top surface requires reconfiguring, capping and revegetation. If feasible, any initial concurrent revegetation of the side slope will also have been implemented.</td><td>Monitoring will take place to confirm success of closure measures implemented at the site (vegetation of side slopes and functionality of waste rock cross walls), until performance objectives and relinquishment/abandonment criteria are met.
Care and maintenance and possible corrective action will be implemented, based on outcomes of monitoring results.
Groundwater quality monitoring will continue to ensure that post-closure water management measures (if any) are adequate, and until satisfactory water quality is achieved.</td></tr>
<tr><td colspan="3">Residue Deposits Associated with Refurbished Infrastructure Areas</td></tr>
<tr><td>Pre-existing ‘fugitive’ or remnant waste material piles will have been consolidated with the new TSF or otherwise appropriately rehabilitated.
Contaminated in-situ soil associated with such areas will already have been excavated and appropriately disposed of where possible.
Any resultant footprint areas will also have been shaped to be free-draining and revegetated.</td><td>Remaining or new waste material will be consolidated with the new TSF prior to rehabilitation.</td><td>Monitoring will take place to confirm success of closure measures implemented at the site.
Care and maintenance and possible corrective action will be implemented, based on outcomes of monitoring results.</td></tr>
</table>

On Last Day of Operations	During Decommissioning and Closure	Post Closure
Shaft Complexes		
• Mining would have ended and Shaft P2 and P5 will become available for rehabilitation. • Geotechnical assessments to determine shaft plugging and capping requirements will have been conducted.	• All remaining shafts will be plugged and capped to an acceptable standard that will ensure human safety in the long run.	• Sealed-off shafts will be monitored for the duration of the aftercare period, to ensure that the plugs are not compromised in any way.
General Surface Rehabilitation		
• Areas from which surface infrastructure had been removed during operations (if applicable) would have been rehabilitated by means of an already proven approach. • Responsibilities for rehabilitation would have been clearly defined in terms of agreements already in place. • Follow-up corrective rehabilitation on poor performing rehabilitation areas would have been conducted. • Vegetation established on rehabilitated and stabilised areas will already be in advanced stages of succession towards the desired stabilisation state, and ongoing rehabilitation studies would be tried and tested towards informing final rehabilitation of the site.	• Appropriate surface rehabilitation approaches established during operational trials will be applied to remaining footprint areas from which infrastructure would be demolished at closure. • Other surface disturbance and footprint areas not already rehabilitated during operations will also be rehabilitated as appropriate. • Initial monitoring and aftercare of rehabilitated areas will be conducted for the duration of the closure plan implementation period to ensure successful vegetation establishment. • Site drainage lines will be reinstated on the rehabilitated surface areas to ensure the site is free draining and to limit or avoid ponding.	• Monitoring will take place to confirm the success of closure measures implemented at the site, until performance objectives and abandonment criteria are met. Surface water, groundwater, and rehabilitation monitoring to be conducted. • Care and maintenance will be implemented and further guided based on monitoring results. • Site relinquishment could be considered based on demonstration of success of the rehabilitation effort.

On Last Day of Operations	During Decommissioning and Closure	Post Closure
Contamination and Waste Management		
• A comprehensive waste management plan will have been compiled and implemented, focusing on: – Negation of accumulation of waste (building rubble, excavation waste etc.) onsite. – The use of dedicated waste cells constructed within the TSF for the disposal of demolition and other waste as necessary, which would have been implemented during operations. – Formalisation of process waste management, especially hazardous waste. – Strategies to remove and/or remediate outlying and extensive contaminated land in-situ, including the possible application of phytoremediation, will have been tested and implemented where possible. – Inventories of reagents and chemicals will be largely run down, and no notable quantities will remain on-site.	• Demolition waste will be decontaminated within the dedicated decontamination bay or area and disposed within the onsite waste cell. Benign concrete waste will also be used for infilling of cavities created by infrastructure demolition. • Any contaminated soils found within the plant area will be excavated and disposed of within the TSF. This could be facilitated in an additional cell. • Any hazardous waste and chemicals (if any) will be transported by road to South Africa and be disposed of at a registered hazardous landfill site.	• Monitoring will take place to confirm success of closure measures implemented at the site, until performance objectives and abandonment criteria are met. • Care and maintenance will be implemented, based on monitoring results. • Site relinquishment will take place upon receipt of a closure certificate.

On Last Day of Operations	During Decommissioning and Closure	Post Closure
Surface and Groundwater Management		
• Surface and groundwater management water on the plant site, and to prevent contaminated water from polluting adjacent water resources. • A robust and functional groundwater model will have been updated throughout operations, and the post-closure water management (and potentially treatment) requirements will already be well understood. • Adequate and representative surface- and groundwater monitoring networks will be in place and operational monitoring data available as baseline against which to compare post-closure water quality results.	• The operational, stormwater management drainage network will be adapted for closure. • The operational storm water drainage network that could be meaningfully utilised for closure could be retained to cater for the temporary remnant surface contamination runoff. • Long-term groundwater management interventions (if required) will be implemented and maintained.	• Once remnant contamination runoff has diminished to acceptable levels, demolish / remove the concrete channels, and rehabilitate to link to overall site drainage. • Monitoring to be conducted to confirm trends documented with operational monitoring and to demonstrate success for site relinquishment. • Care and maintenance will be implemented, based on monitoring results. • Monitoring to be conducted to confirm trends documented with operational monitoring and to demonstrate success for site relinquishment. • Care and maintenance will be implemented, based on monitoring results.

On Last Day of Operations	During Decommissioning and Closure	Post Closure
Socio-economic Aspects		
• A socio-economic plan would have been established, implemented, and well entrenched. The plan would, amongst other aspects, address: – Appropriate stakeholder and community engagement, applying recommendations by specialists to manage or mitigate closure-related stakeholder expectations, most notably with regard to using stabilised areas on site for artisanal mining. – In-depth stakeholder engagement will have been undertaken, informing a realistic next land use for the site, with closure measures aligned to meet the desired next land use. – Socio-economic and community development initiatives and employee re-skilling aimed at facilitating self-sustaining livelihoods and related services for community functioning post closure, as recommended by specialists where applicable. – A community leader would have been appointed by the project to interact with the community regarding commencement of artisanal cropping on stabilised tailings footprint areas.	• Transfer of infrastructure retained for beneficial re-use will be concluded, and any required monitoring thereof during the implementation of the decommissioning and closure activities will be conducted. • Closure-phase community projects such as utilising local labour for appropriate decommission and rehabilitation related actions will take place.	• Appropriate monitoring will take place to confirm success of socio-economic measures, potentially including micro and macro-economic assessments, community surveys, etc. post closure, until performance objectives and abandonment criteria are met. • Implement additional interventions if required.

20.17 KICO Community and Social Activities

KICO have undertaken a number of high-profile community development and cultural activities, including:

- Operation, electricity supply, maintenance, and security of the potable water pump station (this is the single highest cost CR effort, at an estimated $90,000/month).
- Emergency repairs on as-needed basis to the potable water mains reticulation to the municipality.
- Logistics support to the Oral Polio Vaccination (OPV) campaign by the Kipushi Territory Health Zone.
- Annual contributions and attendance at the coronation anniversary of Grand Chief Kaponda of the Lamba tribal group headquartered in Mimbulu village.
- Small animal husbandry and small scale agriculture test plots.
- Bursaries for high performance mathematics and science students in local high schools in Kipushi.
- Student apprenticeships from technical schools in Kipushi, for training in the machine, garage, and welding shops.
- Support to the FIONET malaria diagnostics system implementation, to be installed at 42 health care facilities in the impact Kipushi Health Zone.
- Collaboration with the Municipal authorities on road maintenance, and infrastructure support for municipal buildings.
- Ad hoc school repair programmes.

21 CAPITAL AND OPERATING COSTS

Capital and operating cost estimates have been developed based on the current project costs, the mine and process designs, and discussions with potential suppliers and contractors. The estimated capital costs are to a feasibility level of accuracy and include a contingency of 10%. All monetary figures expressed in this report are in US Dollars ($) unless otherwise stated. Costs have a base date of Q1'21.

21.1 Capital Cost

The total Project direct capital cost estimates are shown in Table 21.1. Capital costs have been estimated separately for each area based on the quantities and design criteria.

The mining costs were applied to the financial model as operating costs or capital costs. In the mining cost model, costs are broken down into specific areas including development, load and haul and production. The KICO budget have been accounted for with the portion in addition to calculated capital included as KICO Holding Budget.

The operating cost summary can be seen in Figure 21.1. The capital cost summary can be seen in Figure 21.2.

The PFS assumed that a mining contractor would be responsible for development and production activities. The FS assumes that all major mining activities will be undertaken on an owner mining basis, with the exceptions of raisebores, which remain a contractor activity. The mining equipment will now be purchased, owned, operated, and maintained by Kipushi Corporation SA (KICO).

The estimating methodology applied in the development of the cost estimates is in line with industry-accepted standards for a Feasibility Study level estimate. The estimated capital cost for the process plant and surface infrastructure accounts for:

- New conveyor connecting Shaft 5 to the process plant run-of-mine (ROM).
- ROM stockpiling.
- New process plant and associated in-plant infrastructure, including laboratories.
- General infrastructure such as electrical substations, motor control centres (MCCs), fuel systems, office buildings, workshops, roads, overhead lines etc.
- Earthworks and terracing.
- Tailings storage facility.

The estimated capital cost was derived from budget quotations received from various equipment suppliers, package pricing for specific areas of the plant and in-house database pricing for minor equipment items.

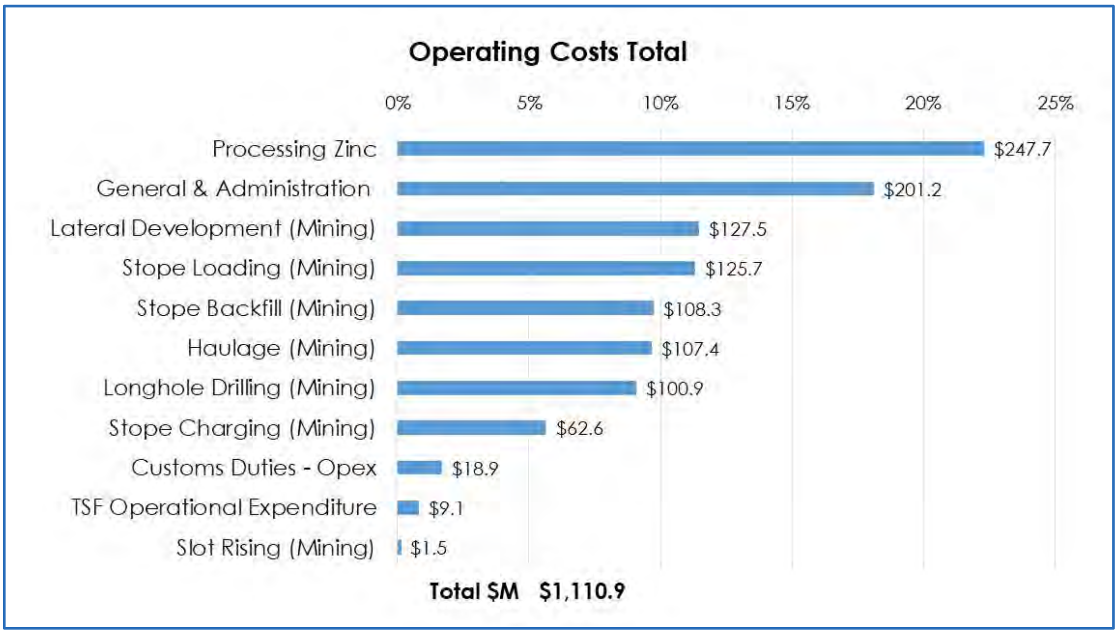
Supply, install, and preliminary and general (P&G) costs by area and by discipline, were factored off the area mechanical equipment supply costs. Earthworks and civils costs were based on geotechnical work, bills of quantities (BOQ), and supply and install rates supplied by contractors based in Lubumbashi.

Table 21.1 Total Project Capital Cost

Item	Pre-production ($M)	Production ($M)	Total ($M)
Mining			
Underground Mining	154	118	272
Capitalised Mining Operating Costs	17	–	17
Subtotal	171	118	289
Process and Infrastructure			
Process and Infrastructure	82	36	117
Road Rehabilitation	1	5	6
Tailings Storage Facility	7	9	15
Capitalised Processing	5	–	5
Subtotal	94	49	143
Closure			
Closure	–	22	22
TSF Closure and Rehabilitation	–	3	3
Subtotal	–	25	25
Indirects and Others			
EPCM	13	–	13
Total Owners Costs	47	–	47
Capitalised General and Administration	9	–	9
Customs Duties	2	3	6
VAT	21	-37	-16
Subtotal	93	-34	59
Capital Cost Before Contingency	357	159	516
Contingency	24	–	24
Capital Cost After Contingency	382	159	540

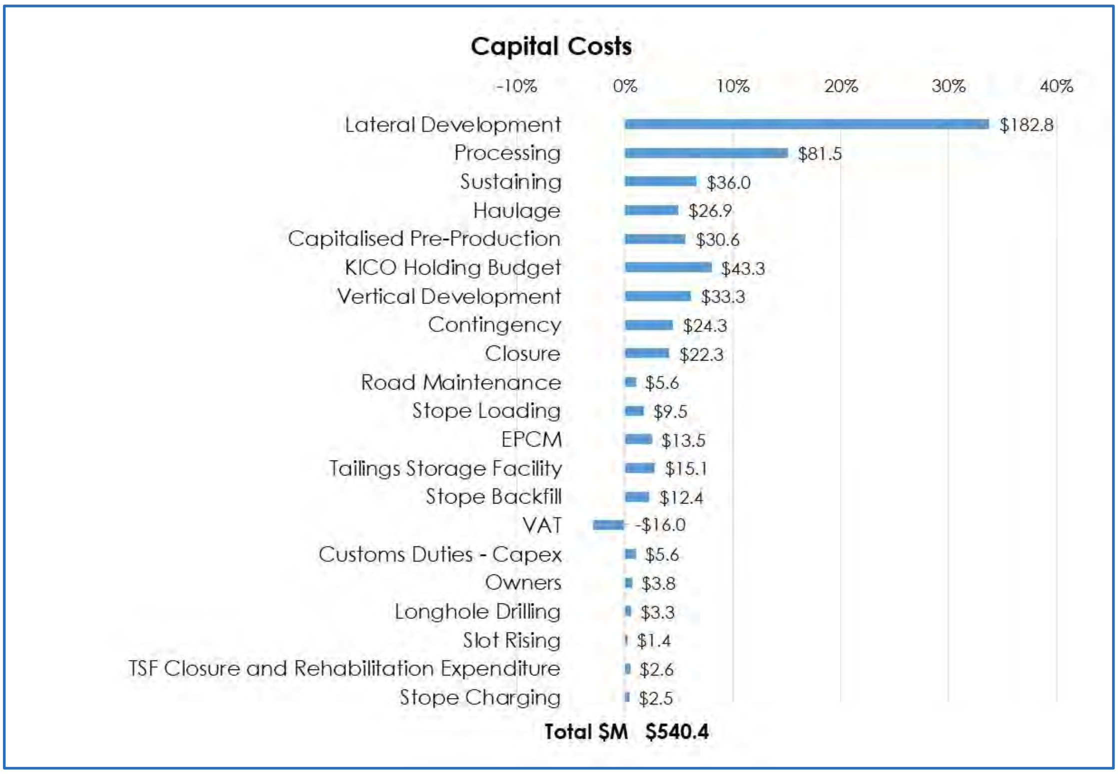
Capital includes only direct project costs and does not include non-cash shareholder interest, management payments, foreign exchange gains or losses, foreign exchange movements, tax pre-payments, or exploration phase expenditure.

Figure 21.1 Operating Cost Summary



OreWin, 2022

Figure 21.2 Capital Cost Summary



OreWin, 2022

The pricing for new buildings was based on budget quotations, whilst the costs for refurbishing buildings was based on BOQs, building supply, and refurbishment rates supplied by contractors local to Lubumbashi and derived from recent projects in the Democratic Republic of Congo (DRC).

21.2 Operating Costs

Operating costs have been estimated from labour numbers and current labour rates, equipment operating costs, consumable and other materials costs, power, fuel, and other estimates. The operating cost estimates have been presented in Table 21.2.

Table 21.2 Estimated Operating Costs

Description	Total ($M)	5-Year Average ($/t Milled)	LOM Average ($/t Milled)
Site Operating Costs:			
Mining	634	61	59
Processing Zn	248	23	23
General and Administration	201	20	19
Tailings Storage Facility	9	1	1
Customs (OPEX)	19	2	2
Total	1,111	107	103

Note: Totals may differ due to rounding.

The operating cost summary can be seen in Figure 21.1.

21.3 Mining Cost Summary

The mining costs were applied to the financial model as operating costs or capital costs. In the mining cost model, costs are broken down into specific areas including development, load and haul, and production. The KICO budget has been accounted for with the portion in addition to calculated capital included as KICO Holding Budget. The operating cost summary can be seen in Figure 21.1. The capital cost summary can be seen in Figure 21.2.

The PFS assumed that a mining contractor would be responsible for development and production activities. The FS assumed that a mining contractor would be responsible for development and production activities for the first three years, after which all major mining activities will be undertaken on an owner mining basis, with the exception of raisebores, which remain a contractor activity. Under owner mining, KICO responsibilities will now include, but not limited to, the decline, level development, stopping, and backfilling, and the operation of the crusher, pumps, and winders. The mining equipment will now be owned, operated, and maintained by KICO employees.

Mining operating costs include:

- Development

- Production
- Load and haul
- Labour
- Main pumping system
- Big Zinc stope pumping
- Other indirect activities
- Backfill

Mining capital costs include:

- Development
- Load and haul
- Labour
- Underground fixed equipment
- Underground mobile equipment
- Office and supply
- KICO Holding Budget
- Mine rehabilitation
- Studies

21.4 Process and Infrastructure Cost Summary

The process and infrastructure were prepared by METC (Technical) Africa Pty Ltd (METC). The estimating methodology applied is in line with industry accepted norms for a FS estimate. The following have been included in the capital costs for process plant cost estimates:

- Ore receiving and crushing
- Dense media separation (DMS)
- Milling
- Flotation
- Concentrate, thickening, filtration, and packaging
- Waste management
- Tailings Storage Facility (TSF)
- Utilities and services
- Reagents
- Plant infrastructure
- Plant mobile equipment

- Spares, first fills and bonds (equipment, reagents, and consumables first fills, commissioning spares)

The following have been included in the capital costs for infrastructure cost estimates:

- Bulk services
- Site preparation
- Buildings and structures (new and refurbished)
- Communications
- IT hardware and software
- Security and access control
- Site Costs
- Mobile equipment
- Services contracts
- Community Support

The following have been included in the operating costs for infrastructure cost estimates:

- Plant consumables
- Crusher Consumables
- Screens
- DMS Cyclones
- Mill Balls – Grinding Media
- Filters
- Packaging Plant Bags
- Plant reagents: FeSi, flocculant, flotation reagents
- Plant mobile equipment
- Plant maintenance
- Power
- Labour
- Production and dispatch
- Plant and infrastructure day work services
- Shift maintenance
- Laboratory service level agreement
- TSF water treatment

The breakdown of the process operating costs can be seen in Table 21.3.

Table 21.3 Processing Costs

Description	$k/month	Distribution
Labour	395.0	26.1%
Power	182.2	12.0%
Water (Free issue)	–	0.0%
Reagent Cost	190.3	12.6%
Consumables – Liners and plant generals	21.5	1.4%
Product Packaging – bulk bags	356.8	23.6%
Mill Grinding Media	53.0	3.5%
Maintenance/Spares Cost	53.4	3.5%
Laboratory	171.8	11.4%
Mobile Equipment	89.6	5.9%
Total Plant	1,513.7	100%

21.5 General and Administration Cost Summary

The General and Administrative (G&A) costs include costs not directly attributable to operational output such as the mining and processing operations. The following costs have been included in total G&A cost:

- Office and general expenses
- Maintenance and inspection contracts
- Equipment and sundry
- Fuels and utilities
- Rentals and leases
- Insurance and insurance taxes
- IT hardware and software
- Personnel transport
- Communications
- Licences and land fees
- Labour
- Accommodation and messing
- Medical support
- Expatriate flights
- Light vehicles
- Environmental, community development and engagement

- Banking and audit fees
- Legal and consultants

21.6 Owners Cost Summary

The owner's costs are 10% of the plant and infrastructure and tailings storage facility capital costs.

21.6.1 Concentrate Transport Costs

The costs for transport from Kipushi via multiple ports (Durban, Walvis Bay, and Dar es Salaam) in South Africa to China (including all taxes) is estimated to total $265.93/t wet concentrate.

This estimate includes the following:

- Handling Mine Site
- Truck Transport to port
- Port Charges
- Ocean Freight –Port to Shanghai
- Logistics Agent Fees
- DRC Government Taxes, Levies, and Duties

21.6.2 Rail Refurbishment Costs

Concentrate is now planned to be trucked from site direct to port. As a result, the rail repair, refurbishment, and construction costs have been removed.

22 ECONOMIC ANALYSIS

22.1 Economic Assumptions

The modelling and taxation assumptions used in the Kipushi 2022 FS are discussed in detail below.

All monetary figures expressed in this report are in US Dollars ($) unless otherwise stated. The Kipushi Project financial model is presented in 2021 constant US dollars, cash flows are assumed to occur evenly during each year and a mid-year discounting approach is taken.

22.1.1 Pricing and Discount Rate Assumptions

The key assumptions in the economic modelling relating to product pricing are tabulated in Table 22.1. A discount rate of 8% is used for calculating net present value.

Table 22.1 Kipushi 2022 FS Key Economic Assumptions

Model Assumption	Unit	Value
Zinc price	$/lb	1.20
Zinc Treatment Charge	$/t concentrate	190
Concentrate Transport to China	$/wmt	266
Zinc pay ability	%	85
DRC NSR Royalty	%	3.50
DRC Royalty / Tax Basis (Conc 51–60% Valorisation)	%	65
DRC Royalty / Tax Basis (Assumed Concentrate Grade)	%	60
DRC Export Taxes	%	1.00
Community Development Contribution	% total revenue	0.30

The estimates of cash flows have been prepared on a real basis as at 1 January 2022 and a mid-year discounting is used to calculate NPV.

In the analysis, carry balances such as tax and working capital calculations are based on nominal dollars and outputs are then deflated for use in the integrated cash flow calculation.

22.1.2 Royalties

The royalty is due upon the sale of the product and is calculated at 3.5% of the gross commercial value of non-ferrous metals.

Gross commercial value is determined by a coefficient depending on the nature of the product, which is assumed to be 65% for zinc concentrate (51–60% Zn content).

22.1.3 Taxation

The DRC Mining Code provides for all the taxes, charges, royalties, and other fees. The key taxes are listed below.

22.1.3.1 General Corporate Taxation

Companies that are the holders of mining rights are subject to corporate income tax (CIT) based on tax at 30% on net income.

A minimum tax of 1% of revenue, which is deductible from the CIT basis, is payable in the event that a mining company is in a loss position or offsetting previous losses carried forwards (see below), and withholding tax on distributions are subject to 10% tax at the shareholder's level. In addition, as from 1 January 2014, the minimum amount of tax payable by mining companies in a year is 1% of the calculated revenue for that specific year.

The mineral products are assumed to be sold on a Free Carrier (FCA) basis. Therefore, KICO would not incur transportation/freight costs, as the buyer would pay for these costs directly.

22.1.3.2 Tax Holidays

The DRC tax legislation does not currently provide for any tax holiday incentives.

22.1.3.3 Tax Losses

Tax losses from a financial year may be deducted from profits earned in subsequent years up to the fifth year following the loss-making period with the use of the carried forward losses limited to 60% of the tax result of the considered year. The aggregate exploration expenditure may be claimed.

22.1.3.4 Research and Development Costs

Research and development costs capitalised during the exploration and construction phases may be amortised over a period of two years from first production, with losses resulting from such an amortisation allowed to be carried forwards. These costs include exploration, owners' costs, certain underground development costs, and interest paid on shareholders loans.

22.1.3.5 Depreciation

Specific mining assets dedicated to mining operations, with useful lives between 4–20 years are depreciated on a straight line basis.

Non-mining assets are depreciated in accordance with the common law. The common low provides different depreciation rates for various assets, e.g. 10 years for plant and equipment.

22.1.3.6 Value Added Tax

Value added tax (VAT) came into effect in the DRC in January 2012. VAT is levied on all supplies of goods and services at a rate of 16% and is not levied on any capital asset movements.

22.1.3.7 VAT Exoneration

Holders of mining rights are currently entitled to exoneration for certain import duties and import VAT for materials and equipment imported for construction of a mine and related infrastructure in accordance with the VAT Act, subject to any unforeseen changes in the law. KICO has successfully received the exoneration in the past in terms of the mining code and it expects to receive such exoneration for most imports for project construction in future in compliance with the mining code.

22.1.3.8 Customs / Import Duties

Imports of equipment are subject to an entry fee at the rate of:

- 2% before the beginning of the mining operations
- 5% until the end of the third year of the first production

All intermediate goods and other consumables are taxed at the rate of 10% of tariffs, except fuels and lubricants for mining activities which are subject to the rate of 5%. The common law system is applicable to all imports after five years from the date of the granting of the mining licence.

22.1.3.9 Export Taxes

Fees and royalties for services rendered – claimed by DRC Agencies and specific public services – is limited to 1.0% of the gross commercial value of the export.

22.1.3.10 Provincial Export Road Tax and Tax on Concentrates

KICO has an agreement in place with the provincial government of Katanga whereby the Kipushi Project has been confirmed to be exempt from the road tax and from the tax on zinc concentrate products.

22.1.3.11 Withholding Taxes

A Withholding tax at the rate of 14% on services supplied by foreign companies established offshore to onshore companies applies. Mining companies are liable for movable property withholding tax at a rate of 10% in respect of dividends and other distributions paid. Non-mining companies are subject to withholding tax of 20%.

22.1.3.12 Dividend Distributions / Interest Repayments

Any dividend distributions will attract a withholding tax of 10%. A withholding tax of 20% applies if the loan is denominated in local DRC currency. If the loan is however denominated in foreign currency no withholding tax is payable. Interest payments to any local intermediate and holding companies attract a withholding tax of 20%.

22.1.3.13 Exceptional Tax on Expatriates

In the DRC, an employer is liable for the exceptional tax on expatriate's remuneration at a rate of 25%. Mining companies are subject to 10%. It is determined in terms of the salaries generated by the work carried out in the DRC and is deductible for purposes of calculating the income tax payable.

22.1.3.14 Tax on excess profits

A special tax on excess profits applies when prevailing commodity prices are more than 25% higher than those prices used in a feasibility study filed with the DRC tax authorities. A tax of 50% is levied on such incremental profits, from which income tax payments are deductible.

22.2 Kipushi 2022 FS Overview and Results

The projected financial results include:

- After-tax NPV at an 8% real discount rate is $941M.
- After-tax internal rate of return (IRR) is 41%.
- After-tax project payback period is 2.3 years.

The estimated C1 cash costs for the first five years of production are $0.63/lb ($1,392/t) zinc and the average for the life of the mine is $0.65/lb ($1,432/t) zinc. Zinc provides the only revenue included in the analysis. There are no credits from other metals included in the cash cost.

22.2.1 Production and Cost Summary

The key results of the Kipushi 2022 FS are summarised in Table 22.2 as compared with the PFS.

The process production forecasts are shown in Table 22.3 and forecast zinc tonnes mined are shown in Figure 22.1. The processing tonnes and concentrate and metal production are summarised in Figure 22.2 and Figure 22.3 respectively.

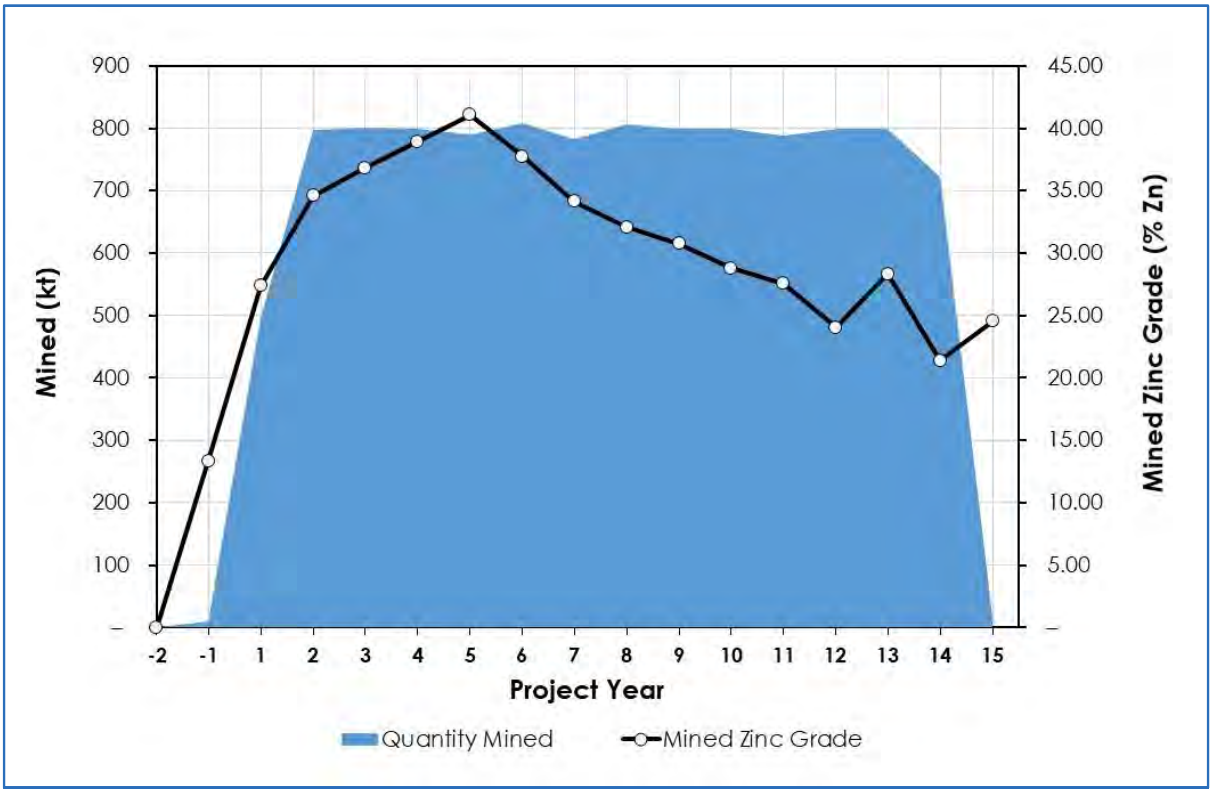
Table 22.2 Kipushi 2022 FS Results Summary

Description	Unit	2022 FS	2017 PFS	% Change
Zinc Feed – Tonnes Processed				
Quantity Zinc Tonnes Treated	kt	10,814	8,581	26%
Zinc Feed grade	%	31.85	32.14	-1%
Zinc Recovery	%	95.63	89.61	7%
Zinc Concentrate Produced	kt (dry)	6,013	4,196	43%
Zinc Concentrate grade	%	54.79	58.91	-7%
Metal Produced				
Zinc	kt	3,294	2,472	33%
Zinc	Mlb	7,263	5,449	33%
Key Cost Results				
Pre-Production Capital	$M	382	337	13%
Peak Funding	$M	393	332	19%
Mine Site Cash Cost	$/lb Payable Zn	0.18	0.16	11%
Transport Costs	$/lb Payable Zn	0.28	0.21	36%
Treatment & Refining Charges	$/lb Payable Zn	0.19	0.15	20%
C1 Cash Costs	$/lb Payable Zn	0.65	0.53	23%
Site Operating Costs	$/t milled	103	88	17%
Zinc Price	$/lb	1.20	1.10	9%
NPV	$M	941	683	38%
Discount Rate	%	8%	8%	0%
IRR	%	41%	35%	16%
Payback Period	Years	2.3	2.2	4%
Project Life	Years	13.8	11.0	25%

Table 22.3 Production Statistics

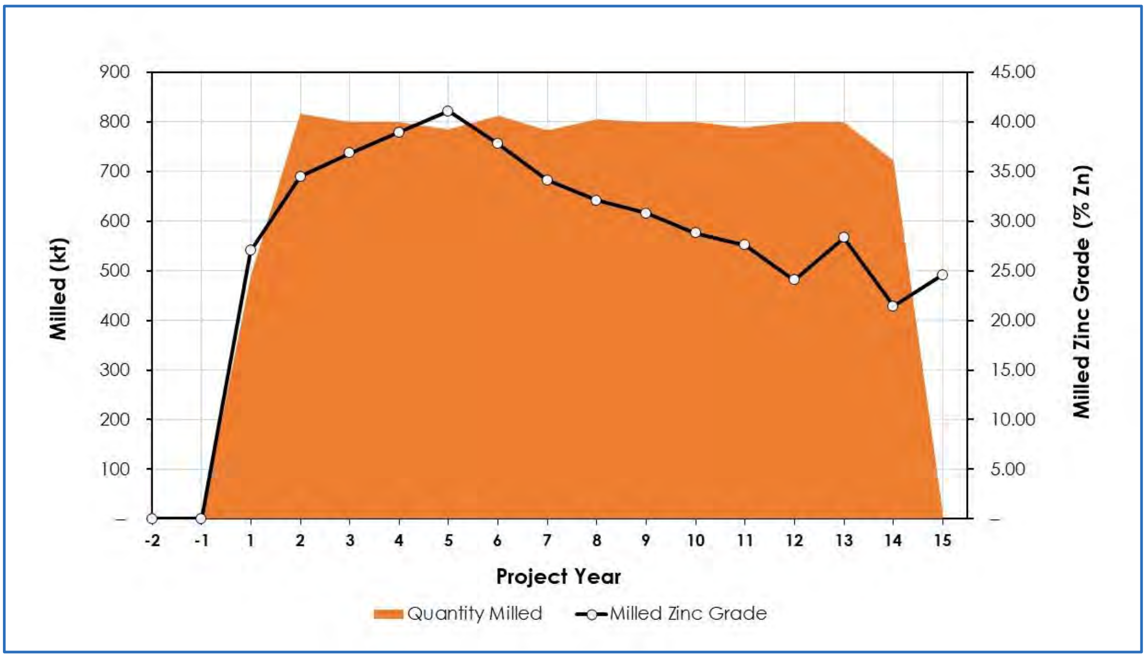
Item	Unit	Total LOM	5-Year Annual Average	LOM Annual Average
Zinc Feed – Tonnes Processed				
Quantity Zinc Tonnes Treated	kt	10,814	792	787
Zinc Feed grade	%	31.85	36.43	31.85
Zinc Recovery	%	95.63	95.87	95.63
Zinc Concentrate Produced	kt (dry)	6,013	508	437
Zinc Concentrate grade	%	54.79	54.79	54.79
Metal Produced				
Zinc	kt	3,294	278	240

Figure 22.1 Zinc Tonnes Mined



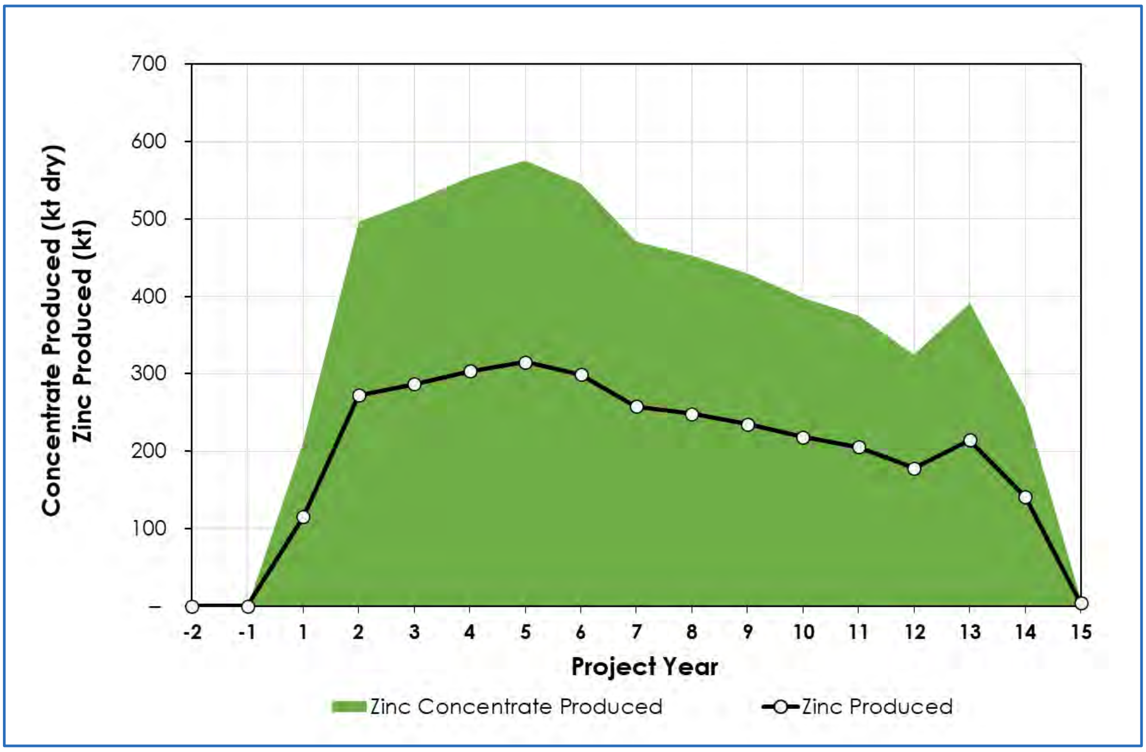
OreWin, 2022

Figure 22.2 Zinc Tonnes Processed



OreWin, 2022

Figure 22.3 Concentrate and Metal Production



OreWin, 2022

22.3 Project Financial Analysis

The estimated Mine site cash costs are shown in Table 22.4. The estimated C1 cash costs for the first five years of production are $0.63/lb ($1,392/t) zinc and the average for the life of the mine is $0.65/lb ($1,432/t) zinc. Zinc provides the only revenue included in the analysis. There are no credits from other metals included in the cash cost. These estimated costs include only direct operating costs of the mine site, namely:

- Mining
- Concentration
- Tailings
- General and administrative (G&A) costs
- Government fees and charges (excluding corporate taxation)

The projected financial results include:

- After-tax net present value (NPV) at an 8% real discount rate is $941M.
- After-tax internal rate of return (IRR) is 41%.
- After-tax project payback period is 2.33 years.

Table 22.4 Cash Costs

Description	5-Year Average ($/lb Zn)	LOM Average ($/lb Zn)
Mine Site Cash Cost	0.16	0.18
Transport Costs	0.28	0.28
Treatment and Refining Charges	0.19	0.19
C1 Cash Cost	0.63	0.65
Royalties	0.07	0.07
Total Cash Costs	0.70	0.72

The estimated revenues and operating costs have been presented in Table 22.5, along with the estimated net sales revenue value attributable to each key period of operation. The analysis uses price assumptions of $1.20/lb ($2,646/t) Zn. The prices are based on a review of consensus price forecasts from a financial institutions and similar studies that have recently been published. The estimated total Project direct capital costs are shown in Table 22.6.

Table 22.5 Operating Costs and Revenues

Description	Total ($M)	5-Year Average ($/t Milled)	LOM Average ($/t Milled)
Revenue			
Gross Sales Revenue	7,408	790	685
Less Realisation Costs			
Transport Costs	1,757	187	162
Treatment and Refining Charges	1,142	122	106
Royalties	405	43	37
Total Realisation Costs	3,305	352	306
Net Sales Revenue	4,103	438	379
Less Site Operating Costs			
Mining	634	61	59
Processing Zn	248	23	23
General and Administration	201	20	19
Tailings Storage Facility	9	1	1
Customs (OPEX)	19	2	2
Total	1,111	107	103
Operating Margin ($M)	2,992	331	277
Operating Margin (%)	40	42	40

Table 22.6 Total Project Capital Cost

Item	Pre-production ($M)	Production ($M)	Total ($M)
Mining			
Underground Mining	154	118	272
Capitalised Mining Operating Costs	17	–	17
Subtotal	171	118	289
Process and Infrastructure			
Process and Infrastructure	82	36	117
Road Rehabilitation	1	5	6
Tailings Storage Facility	7	9	15
Capitalised Processing	5	–	5
Subtotal	94	49	143
Closure			
Closure	–	22	22
TSF Closure and Rehabilitation	–	3	3
Sub-total	–	25	25
Indirects and Others			
EPCM	13	–	13
Total Owners Costs	47	–	47
Capitalised General and Administration	9	–	9
Customs Duties	2	3	6
VAT	21	-37	-16
Sub-total	93	-34	59
Capital Cost Before Contingency	357	159	516
Contingency	24	–	24
Capital Cost After Contingency	382	159	540

Capital includes only direct project costs and does not include non-cash shareholder interest, management payments, foreign exchange gains or losses, foreign exchange movements, tax pre-payments, or exploration phase expenditure.

The projected financial results for undiscounted and discounted cash flows at a range of discount rates, IRR, and payback are shown in Table 22.7. The key economic assumptions for the analysis are shown in Table 22.8.

Table 22.7 Financial Results

	Discount Rate	Before-tax	After-tax
Net Present Value ($M)	Undiscounted	2,452	1,946
	5.00%	1,536	1,228
	8.00%	1,175	941
	10.00%	986	790
	12.00%	829	663
	15.00%	640	510
	18.00%	495	391
	20.00%	415	326
Internal Rate of Return	–	44%	41%
Project Payback Period (Years)	–	2.3	2.3

Table 22.8 Economic Assumptions

Parameter	Unit	Financial Analysis Assumption
Zinc Price	$/lb	1.20
Zinc Treatment Charge	$/t concentrate	190

The results of NPV sensitivity analysis to a range of zinc prices, discount rates, treatment charges and cost are shown in Table 22.9 to Table 22.11. The sensitivity analysis demonstrates how robust the after-tax NPV is with respect to changes in key economic factors. A 20% increase in site operating costs shows a 9% reduction in the after-tax NPV.

The estimated cumulative cash flow is depicted in Figure 22.4 and a complete cash flow is provided in Table 22.12.

Table 22.9 After-Tax NPV8% Sensitivity to Zinc Price and Discount Rates

Discount Rate (%)	Zinc ($/lb)								
	0.80	1.00	1.10	1.20	1.30	1.40	1.60	1.80	2.00
Undiscounted	33	1,166	1,563	1,946	2,338	2,734	3,901	5,458	6,954
5%	-73	678	962	1,228	1,495	1,763	2,540	3,566	4,538
8%	-115	486	722	941	1,159	1,376	1,999	2,817	3,586
10%	-136	385	595	790	982	1,172	1,715	2,424	3,088
12%	-153	302	490	663	833	1,002	1,478	2,097	2,674
15%	-172	203	363	510	654	795	1,191	1,701	2,175
18%	-186	127	265	391	514	634	966	1,392	1,787
20%	-193	87	211	326	437	546	844	1,224	1,576

Table shows NPV8% $M.

Table 22.10 After-Tax NPV8% and IRR Sensitivity to Zinc Price and Zinc Treatment Charge

Zinc Treatment Charge ($/t)	Zinc Price ($/lb)								
	0.80	1.00	1.10	1.20	1.30	1.40	1.60	1.80	2.00
100	160	702	922	1,140	1,357	1,574	2,202	3,022	3,830
	15%	33%	40%	47%	53%	59%	73%	88%	101%
130	67	635	855	1,074	1,291	1,508	2,135	2,954	3,757
	11%	31%	38%	45%	51%	57%	71%	86%	100%
160	-26	567	788	1,007	1,225	1,442	2,065	2,885	3,678
	7%	28%	36%	43%	49%	55%	70%	85%	98%
190	-115	486	722	941	1,159	1,376	1,999	2,817	3,586
	1%	26%	34%	41%	47%	54%	68%	83%	97%
220	-206	400	654	874	1,093	1,310	1,932	2,748	3,494
	N/A	23%	31%	39%	45%	52%	66%	82%	95%
250	-300	307	587	808	1,026	1,244	1,866	2,679	3,402
	N/A	20%	29%	37%	44%	50%	65%	80%	94%
300	-444	152	454	696	916	1,134	1,756	2,562	3,249
	N/A	15%	25%	33%	40%	47%	62%	77%	91%

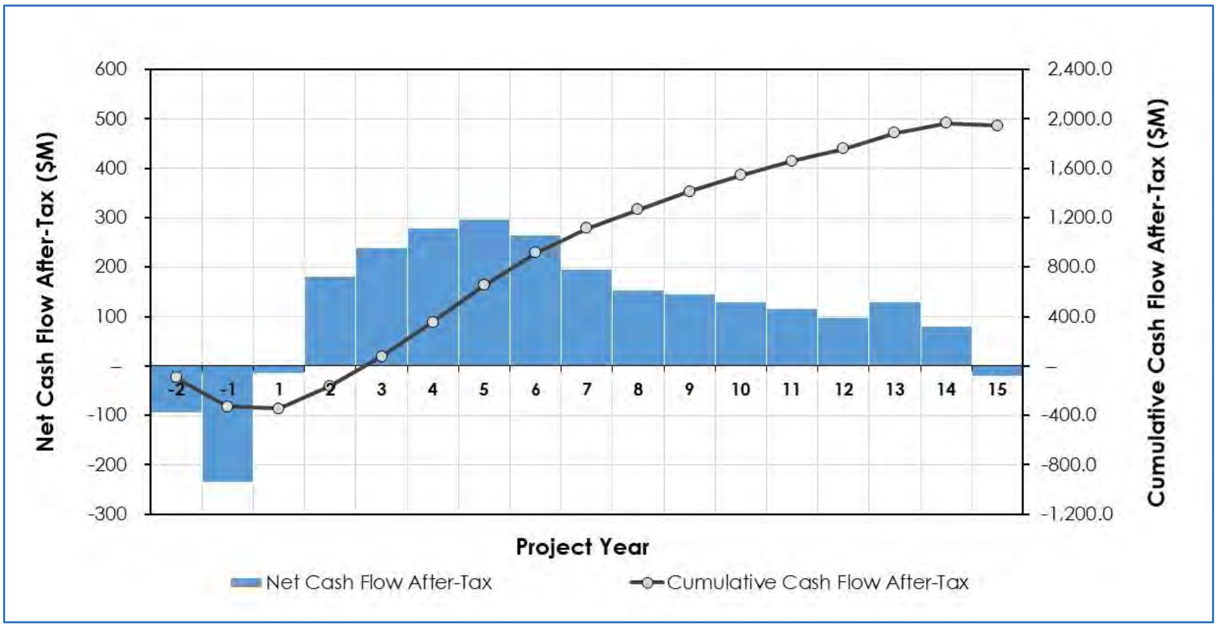
Table shows NPV8% $M and IRR.

Table 22.11 After-Tax NPV8% Sensitivity to Operating and Capital Cost

Cost Item	Change in Cost								
	-30%	-20%	-10%	-5%	–	+5%	+10%	+20%	+30%
Site Operating Cost	1,068	1,025	983	962	941	920	899	856	814
Capital Cost	1,036	1,004	972	957	941	925	909	878	847

Table shows NPV8% $M.

Figure 22.4 Cumulative Cash Flow



OreWin, 2022

Table 22.12 Estimated Cash Flow

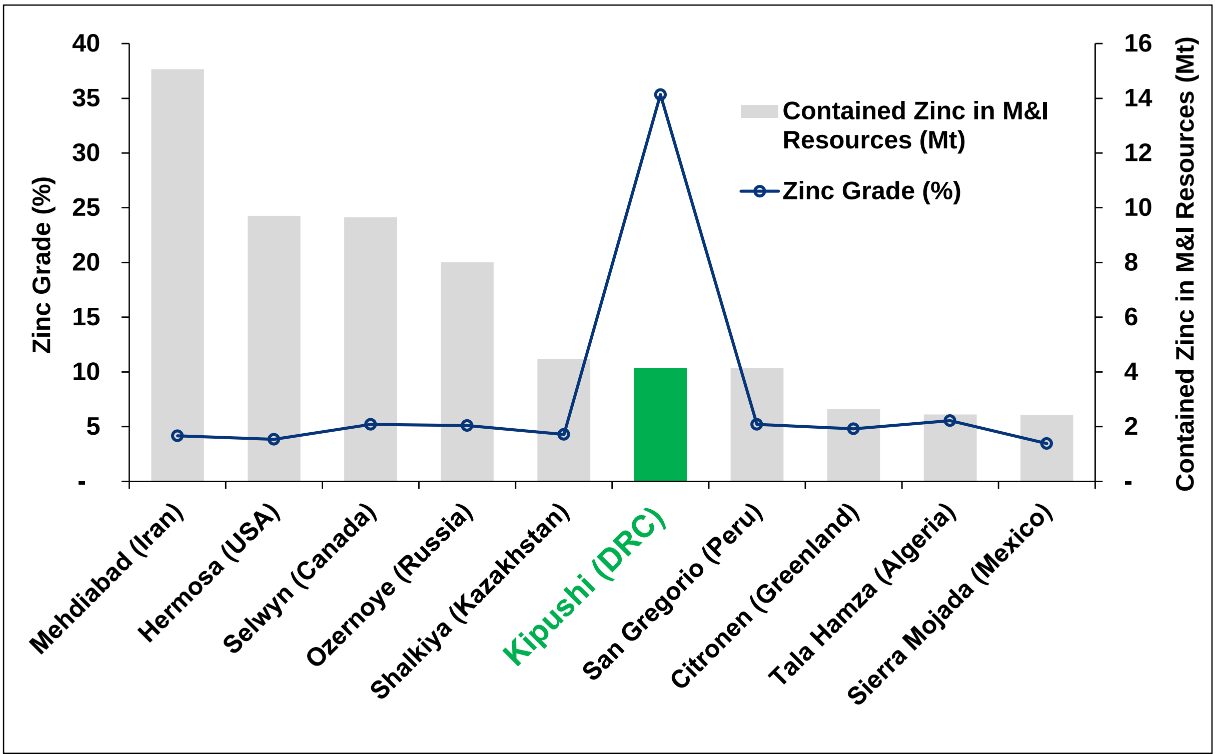
Description	Unit	Total	Period																
			–2	–1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Total Gross Revenue	$M	7,408	–	–	259	612	645	683	709	672	580	558	529	491	462	401	482	317	8
Total Realisation Costs	$M	3,305	–	–	116	273	288	305	316	300	259	249	236	219	206	179	215	141	4
Net Revenue	$M	4,103	–	–	144	339	357	378	393	372	321	309	293	272	256	222	267	176	5
Site Operating Costs																			
Total Mining	$M	651	0	4	52	54	42	46	46	47	50	51	45	49	40	42	38	41	3
Processing Zn	$M	252	1	2	12	18	18	18	18	18	18	18	18	18	18	18	18	17	1
General and Administration	$M	210	1	5	15	17	16	15	14	14	14	14	14	14	14	14	14	13	3
TSF Operational Expenditure	$M	10	–	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	–
Capitalised Pre-Production	$M	-31	-2	-12	-16	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Customs Duties – Opex	$M	19	0	0	1	1	1	1	1	2	2	2	1	2	1	1	1	1	0
Total Operating Costs	$M	1,111	-0	0	65	91	78	81	80	82	85	85	79	84	74	76	72	73	7
Operating Surplus or Deficit	$M	2,992	0	-0	78	248	279	298	313	291	236	224	214	188	182	146	195	103	-2
Capital Costs																			
Mining Capital Cost	$M	272	43	89	45	48	7	1	1	2	4	2	6	3	12	6	2	1	–
Processing	$M	82	7	71	3	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Road Maintenance	$M	6	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Tailings Storage Facility	$M	15	4	3	–	–	4	5	–	–	–	–	–	–	–	–	–	–	–
Sustaining	$M	36	–	–	2	3	3	3	3	3	3	3	3	3	3	3	3	3	0
KICO Holding Budget	$M	43	27	16	1	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Closure	$M	25	–	–	–	–	–	–	0	0	0	0	0	–	–	–	–	1	24
EPCM	$M	13	2	11	1	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Owners	$M	4	0	3	1	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Contingency	$M	24	5	16	3	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Capitalised Pre-Production	$M	31	2	12	16	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Customs Duties – Capex	$M	6	0	2	1	2	0	0	0	0	0	0	0	0	0	0	0	0	–
Working Capital	$M	0	–	–	-7	-3	-0	-1	-3	2	6	-1	0	1	-1	2	-4	10	1
VAT	$M	16	-4	-13	-13	-8	-2	20	24	9	0	-0	-0	0	0	0	0	-0	4
Total Capital	$M	572	87	210	52	41	11	28	24	17	13	4	9	7	15	11	1	14	28
Cash Flow Before-Tax	$M	2,420	-87	-210	26	207	268	270	288	274	223	220	205	181	167	135	194	89	-31
Tax Payable	$M	506	–	–	1	2	24	29	32	31	39	63	59	53	49	39	55	28	–

22.4 Comparison to Other Projects

Using data for other zinc projects provided by Wood Mackenzie comparisons with the Kipushi 2022 FS were made for the following results: contained zinc in Measured and Indicated Resource, production, capital intensity, and C1 Cash Costs.

The Kipushi Project Mineral Resource Estimate, June 2018 includes Measured and Indicated Resources of 11.8 Mt at 35.34% Zn. This grade is more than twice as high as the Measured and Indicated Mineral Resources of the world's next-highest-grade zinc project, according to Wood Mackenzie, a leading, international industry research and consulting group (Figure 22.5).

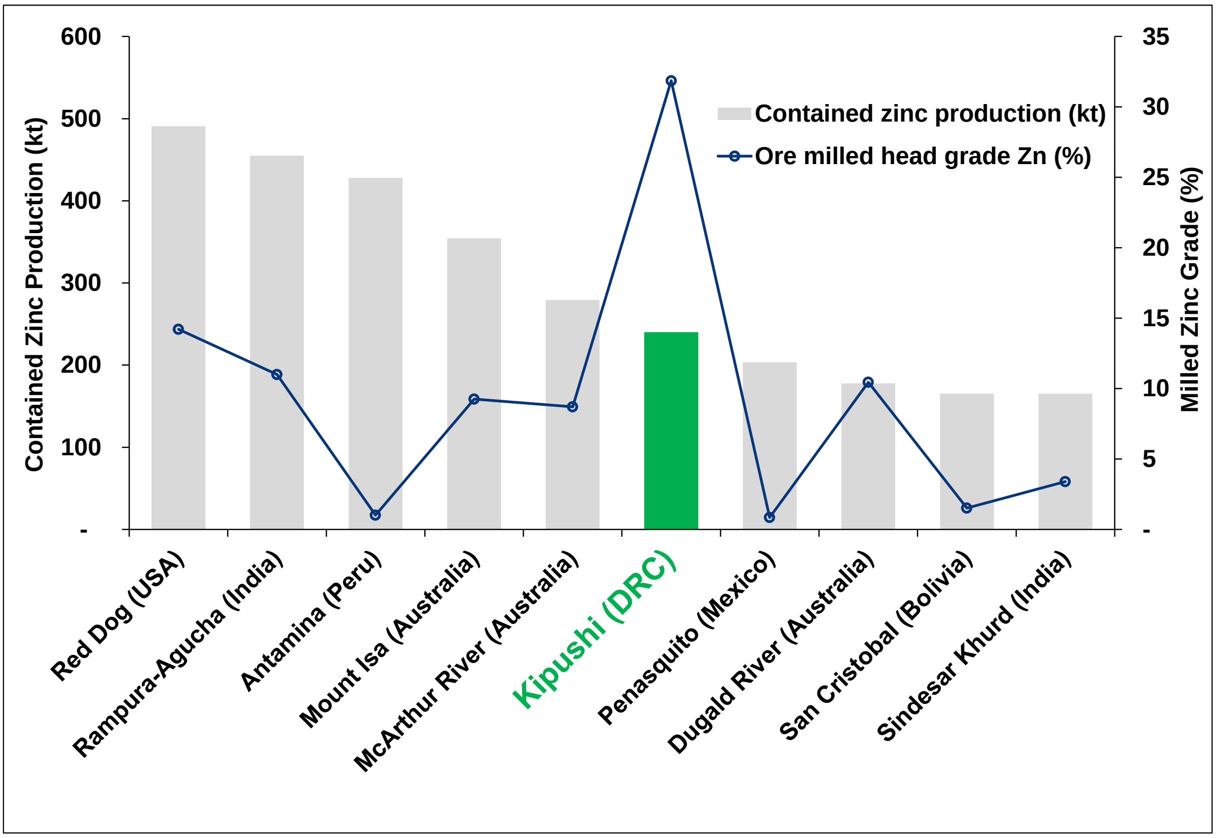
Figure 22.5 Top 10 Zinc Projects by Contained Zinc



Wood Mackenzie, 2022

Life-of-mine (LOM) average planned zinc concentrate production of 437 ktpa, with a concentrate grade of 54.8% Zn, is expected to rank the Kipushi Project, once in production, among the world's major zinc mines (Figure 22.6). Based on research by Wood Mackenzie the world's major zinc mines defined as the world's 10 largest zinc mines ranked by production in 2020.

Figure 22.6 Major Zinc Mines Estimated 2020 Annual Zinc Production and Grade



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Kipushi's estimated low capital intensity relative to comparable 'probable' and 'base case' zinc projects identified by Wood Mackenzie is highlighted in Figure 22.7. The figure uses comparable projects as identified by Wood Mackenzie, based on public disclosure and information gathered in the process of Wood Mackenzie's research.

Based on comparative data from Wood Mackenzie, C1 cash cost of $0.65/lb of zinc is expected to rank the Kipushi Project, once in production, in the second quarter of the 2020 cash cost curve for zinc producers globally.

Figure 22.8 represents C1 cash costs which reflect the direct cash costs of producing paid metal incorporating mining, processing and offsite realisation costs having made appropriate allowance for the co-product revenue streams. Based on public disclosure and information gathered in the process of Wood Mackenzie's research.

Figure 22.7 Capital Intensity for Zinc Projects

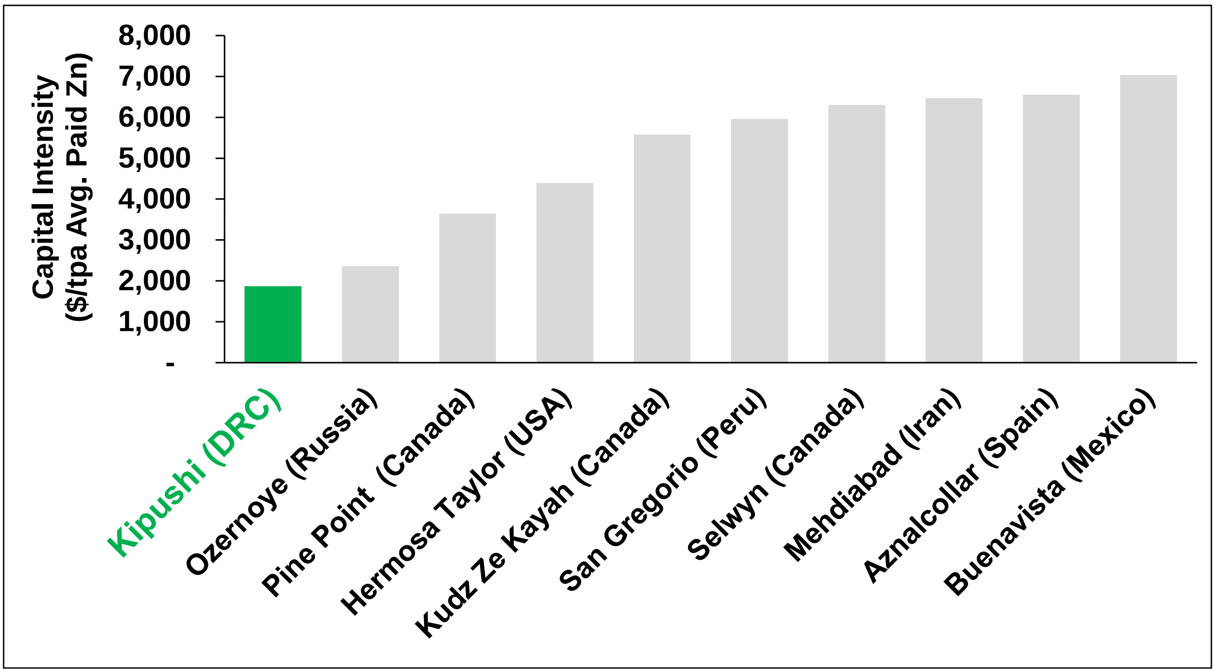

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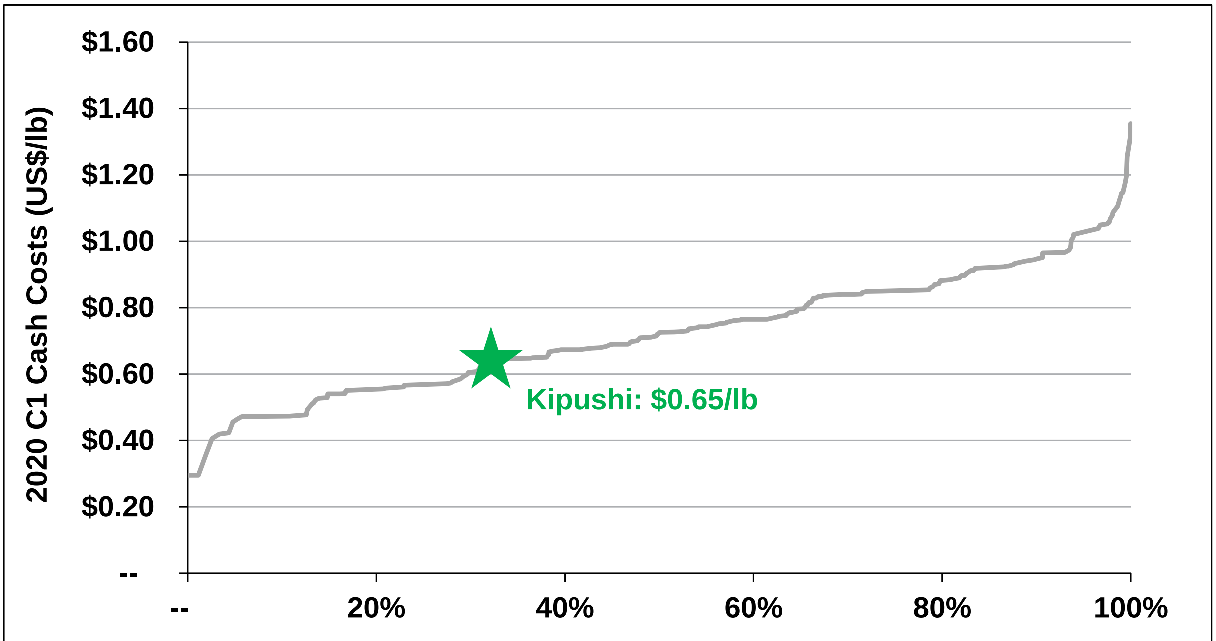
Figure 22.8 2020 C1 Cash Costs



Wood Mackenzie, 2022

23 ADJACENT PROPERTIES

This section not used.

24 OTHER RELEVANT DATA AND INFORMATION

This section not used.

25 INTERPRETATION AND CONCLUSIONS

25.1 Mineral Resource Estimate

Mineral Resources for the Project have been estimated using drill core data, have been performed using industry best practices (CIM, 2003), and conform to the requirements of CIM Definition Standards (2014). The MSA Group (Pty) Ltd (MSA) considers the Kipushi resource model to be suitable to support feasibility level mine planning.

Areas of uncertainty that may materially impact the Mineral Resource estimates include:

- Assumptions used to generate the data for consideration of reasonable prospects of eventual economic extraction for the Kipushi deposit.
- Metallurgical recovery assumptions in the presence of pyrite or iron-rich zones.
- Exploitation of the Kipushi Project requires rehabilitation of existing mine infrastructure, building of a new processing facility, and rehabilitation or building of transportation infrastructure. Changes in the assumptions as to operating and capital costs associated with the proposed development may affect the base case cut-off grades selected for the Kipushi Mineral Resource estimates.
- Commodity prices and exchange rates.

25.2 Mineral Reserve Estimate

Mineral Reserves for the Kipushi 2022 FS conform to the requirements of CIM Definition Standards (2014). Mining Plus have utilised development processes and cost estimates to the level of accuracy required to state reserves and support a feasibility-level study.

Areas of uncertainty that may impact the Mineral Reserve Estimate include:

- Commodity prices and exchange rates.
- The continuity and dip of the ore will need to be better defined prior to and during the mining stages.
- The amount of groundwater present in the orebody during the mining cycle.

25.3 Kipushi 2022 Feasibility Study

The Kipushi 2022 FS identified a positive business case and recommended that the Kipushi Project continue with the preparatory programmes.

25.4 Metallurgical Testing and Process Design

The Kipushi FS testwork programme has provided a set of realistic outcomes that have informed the process flowsheet development and the predictions of likely operational scenarios.

Based on metallurgical test results and interpretations, it is anticipated that a saleable zinc sulfide flotation concentrate at a grade exceeding 53% Zn could be produced at an overall 90% Zn recovery from a ROM ore feed with an average zinc grade of 32% Zn. Extensive metallurgical testing has resulted in a conventional zinc sulfide process flowsheet involving dense media separation (DMS) and flotation methods. The ore will be crushed to 12 mm top particle size prior to feeding the DMS operating at –12 mm + 1 mm feed size fraction. DMS product and –1 mm fines material is milled to 80% passing 106 µm in the ball mill and transferred to the rougher flotation circuit where final zinc concentrate is produced.

This FS assessment review establishes the Kipushi Project as a robust, high-grade zinc deposit that should proceed with project development. Highlights of this study include:

- The study defines an undeveloped significant, massive zinc sulfide ore deposit, refer to as the Big Zinc orebody.
- Robust, proven metallurgical process producing highly marketable zinc concentrate.
- DMS testing on –12 mm + 1 mm ore feed with an average ore grade composition resulted in zinc stage recovery of greater than 98%.
- Bulk flotation testing on combined crusher fines and DMS sinks, from ore feed with an average ore grade composition, resulted in a saleable zinc concentrate product at zinc grade greater than 53% and flotation stage recovery of greater than 96%.
- Robust processing plant layout design, allowing safety practices and flexible operability.

Substantial infrastructure at the Kipushi site already exist, including access road networks, power and water supply systems.

26 RECOMMENDATIONS

26.1 Further Assessment

The Kipushi 2022 FS identified a positive business case and recommended that the Kipushi Project continue with the preparatory and early works programmes. There are a number of areas that require further examination and study and arrangements that need to be put in place to advance the development of the Kipushi Project.

The results of the Kipushi 2022 FS suggest that further study should be undertaken. In particular, the investigation of logistics and transport, mining, and processing.

26.2 Drilling

No further drilling is recommended prior to the commencement of mining.

26.3 Underground Mining

The following is a list of mining recommendations for the Project:

- Complete preparatory work for the early works as soon as practicable.
- Trade-off alternative options for the backfilling strategy.
- Investigate potential sources of waste to meet the backfill shortfall.
- Complete hydrological studies and data evaluation to better determine impacts on underground mining conditions and productivities at Kipushi.
- Stress measurements during the execution stage of the project.
- The performance of cemented rock fill (CRF) in stopes will need to be investigated through trials in the early stopes.
- Drill geotechnical holes to determine ground conditions at each ventilation raise.
- Determine the virgin rock temperature gradient.
- Develop an operating philosophy to optimise waste rock movements.
- Perform a detailed simulation of the underground traffic flow at peak production.
- Conduct a survey of the local workforce to determine available skill levels. The mining productivities and costs have assumed that skilled tradesmen are available to fill the critical mine operational positions.

26.4 Process Plant

The following is a list of process recommendations for the Kipushi deposit:

- Kipushi Corporation SA should develop a reliable and economic measurement method to estimate the zinc mineralogy of samples and plant feed. This will assist in the prediction of concentrate grades (including copper and lead contents) and zinc recoveries.

- The flotation circuit is designed to handle a specific range of ore head grades, correlating to a design range of concentrate mass pull. Significant changes in flotation plant mass pull, due to high or low flotation feed zinc head grades, would impact flotation performance and potentially deviate slurry flowrates outside of the design capabilities of the installed equipment. As such, ore feed blending must ensure feed grade changes are limited in the flotation section.
- A further review of concentrate quality control measures in the design is warranted once specific demands of customers regarding impurities such as copper and lead are understood.

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