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Kipushi Zn–Cu Project

Kipushi 2016
Preliminary Economic Assessment

May 2016

Job No. 15006

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Title Page

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Democratic Republic of the Congo

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Qualified Persons:

Bernard Peters, B. Eng. (Mining), FAusIMM (201743), employed by OreWin as Technical Director - Mining was responsible for: Sections 1.1, 1.2, 1.6, 1.8 to 1.12, 1.13; Section 2; Section 3; Section 4 to 5; Section 13; Section 15 to 22; Section 23 to 24; Section 25; Section 26.1; Section 27.

Michael Robertson, BSc Eng (Mining Geology), MSc (Structural Geology), Pr.Sci.Nat SACNASP, MGSSA, MSEG, MSAIMM, employed by MSA as a Principal Consulting Geologist was responsible for: Section 1.3 to 1.5, 1.13; Section 2; Section 3; Section 6 to Section 12; Section 25; Section 26.2; Section 27.

Jeremy Witley, BSc Hons (Mining Geology), MSc (Eng), Pr.Sci.Nat SACNASP, FGSSA, employed by MSA as a Principal Resource Consultant was responsible for: Section 1.7, 1.13; Section 2; Section 3; Section 14; Section 25; Section 26.2; Section 27.

Signature Page

Project Name:	Kipushi Zn–Cu Project
Title:	Kipushi 2016 Preliminary Economic Assessment
Location:	Haut-Katanga Province Democratic Republic of the Congo
Effective Date of Technical Report:	12 May 2016

/s/ Bernard Peters
Date of Signing:
Bernard Peters
Technical Director – Mining
OreWin Pty Ltd
BEng (Mining), FAusIMM (201743)

/s/ Michael Robertson,
Date of Signing:
Michael Robertson
Principal Consulting Geologist
The MSA Group (Pty) Ltd
BSc Eng (Mining Geology), MSc (Structural Geology), Pr.Sci.Nat SACNASP, MGSSA, MSEG, MSAIMM

/s/ Jeremy Witley,
Date of Signing:
Jeremy Witley
Principal Resource Consultant
The MSA Group (Pty) Ltd
BSc Hons (Mining Geology), MSc (Eng), Pr.Sci.Nat SACNASP, FGSSA

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1 SUMMARY

1.1 Introduction

The Kipushi Zn-Cu Project is located adjacent to the town of Kipushi in the south–western part of the Haut-Katanga Province in the Democratic Republic of the Congo (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 subsidiary of Ivanhoe Mines Ltd. (Ivanhoe)) and La Générale des Carrières et Des Mines (Gécamines) have a joint venture agreement (JV Agreement) over the Kipushi Zn-Cu Project. Ivanhoe and Gécamines respectively own 68% and 32% of the Kipushi Zn-Cu Project through Kipushi Corporation SA (KICO), the mining rights holder of the Kipushi Zn-Cu Project.

The JV Agreement was signed on 14 February 2007 and established KICO for the exploration, development, production, and product marketing of the Kipushi Zn-Cu 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.

The Kipushi 2016 Preliminary Economic Assessment (Kipushi 2016 PEA) has been prepared for Ivanhoe by OreWin and MSA and presents the results of exploration drilling, mineral resource estimation, and mine planning on the Big Zinc zone for the redevelopment of the Kipushi Zn-Cu Project.

Kipushi is connected to Lubumbashi by a paved road. The closest public airport to the Kipushi Zn-Cu Project is at Lubumbashi where there are daily domestic, regional and international scheduled flights.

The Kipushi mine, which was placed on care-and-maintenance in 1993, flooded in early 2011 due to a lack of pumping maintenance over an extended period. Water reached 862 m below surface at its peak. Following dewatering and access to the main working level in December 2013, a 25,400 m underground drilling programme was carried out by KICO starting in March 2014 and continuing through November 2015. The drilling was primarily designed to confirm and update Kipushi's Historical Estimate for the Big Zinc zone and to further expand these resources along strike and at depth. Where infrastructure permitted, drilling also targeted some of the copper-rich zones in the Série Récurrente and Copper Nord Riche zones. This drilling was limited in extent and only occurred below the 1,150 mRL level. At the data cut-off date of 16 December 2015, a total of 97 holes had been drilled for 25,419 m including 51 holes that tested the Big Zinc zone.

The Mineral Resource prepared for the Kipushi 2016 PEA estimate includes Measured and Indicated Resources of 10.2 Mt at 34.89% Zn and 0.65% Cu and Inferred Resources of 1.9 Mt at 28.24% Zn and 1.18% Cu.

Underground mining of the Big Zinc zone is planned to be undertaken using a Sublevel Open Stoping (SLOS) method. The mine production is expected to be 1.1 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 processed in a dense media separation (DMS) plant.

Life-of-mine average planned zinc concentrate production is anticipated to be 530,000 dry tpa, with a concentrate grade of 53% Zn.

Concentrate is planned to be transported by rail directly from Kipushi Station to the port of Durban in South Africa, from there it would be shipped by sea to customers.

Total zinc production is anticipated to be 9.4 Mt at 32.15% Zn over a period of ten years to produce 2,807 kt zinc metal in concentrate. In addition to the zinc, total copper production is anticipated to be 0.5 Mt at 5.41% Cu to produce 27 kt of copper metal in concentrate. Copper is planned to be treated under a toll treatment arrangement.

The estimates of cash flows have been prepared on a real basis as at 1 January 2016 and a mid-year discounting is used to calculate Net Present Value (NPV). All monetary figures expressed in this report are US dollars (US$) unless otherwise stated.

The economic analysis uses price assumptions of $2,227/t Zn and $6,614/t Cu. The prices are based on a review of consensus price forecasts from a financial institutions and similar studies recently published.

The projected financial results include:

- After-tax net present value (NPV) at an 8% real discount rate is $533M
- After-tax internal rate of return (IRR) is 30.9%
- After-tax project payback period is 2.2 years

The key results of the Kipushi 2016 PEA are summarised in Table 1.1.

Table 1.1 Kipushi 2016 PEA Results Summary

Item	Unit	Total
Zinc Feed - Tonnes Processed		
Quantity Zinc Tonnes Treated	kt	9,394
Zinc Feed grade	%	32.15
Zinc Recovery	%	92.94
Zinc Concentrate Produced	kt (dry)	5,296
Zinc Concentrate grade	%	53.00
Copper Feed - Tonnes Processed		
Quantity Copper Tonnes Treated	kt	547
Copper Feed grade	%	5.41
Copper Recovery	%	90.00
Copper Concentrate Produced	kt (dry)	106
Copper Concentrate grade	%	25.00
Metal Produced		
Zinc	kt	2,807
Copper	kt	27
Key Cost Results		
Pre-Production Capital	$M	409
Mine Site Cash Cost	$/lb Zn	0.12
Realisation	$/lb Zn	0.44
Total Cash Costs After Credits	$/lb Zn	0.54
Site Operating Costs	$/t milled	74.77

The key economic assumptions for the analyses are shown in Table 1.2.

Table 1.2 Metal Prices and Terms

Parameter	Unit	Financial Analysis Assumption
Zinc Price	$/t	2,227
Copper Price	$/t	6,614
Zinc Treatment Charge	$/t concentrate	200.00
Copper Treatment Charge	$/t concentrate	90.00
Copper Refining Charge	$/t Cu	198.42

The projected financial results for undiscounted and discounted cash flows at a range of discount rates, internal rate of return (IRR) and payback are shown in Table 1.3.

The results of NPV sensitivity analysis to a range of zinc prices and discount rates is shown in Table 1.4.

Table 1.3 Financial Results

	Discount Rate	Before Taxation	After Taxation
Net Present Value ($M)	Undiscounted	1,473	1,076
	5.0%	973	696
	8.0%	759	533
	10.0%	642	444
	12.0%	542	368
Internal Rate of Return	–	36.4%	30.9%
Project Payback Period (Years)	–	2.1	2.2

Table 1.4 After Tax Zinc Price Sensitivity – Discount Rates

Discount Rate	**Zinc ($/t)**						
	1,500	**1,750**	**2,000**	**2,227**	**2,500**	**2,750**	**3,000**
Undiscounted	-157	325	719	1,076	1,507	1,901	2,295
5%	-210	146	436	696	1,008	1,293	1,577
8%	-230	69	315	533	794	1,032	1,269
10%	-240	28	249	444	677	889	1,101
12%	-248	-7	193	368	577	767	957

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 to April) with an annual average rainfall of 1,208 mm. Average annual maximum and minimum temperatures are 28°C and 14°C respectively.

A large proportion of the local population was employed at the mine until the suspension of mining operations in 1993. A number of mine personnel have been retained to keep the mine secure and many of these people still live in the area. As of 31 December 2014, KICO employed approximately 400 people.

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

1.3 Drilling Programmes

1.3.1 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 zone and 12.5 m apart along the Série Récurrente zone, 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 Kipushi 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 zone. A total of 84 holes intersected the Big Zinc zone, 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 mRL which intersected the Big Zinc zone 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.

1.3.2 KICO Drilling

All work carried out during the KICO underground drilling campaign was performed according to documented standard operating procedures for the Kipushi Zn-Cu Project.

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. 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 zones. Along the section lines, the drillholes intersected mineralisation between 10–50 m apart within the Big Zinc zone 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.

At the cut-off date of 16 December 2015 for data, a total of 97 holes had been drilled for 25,419 m including 51 holes that tested the Big Zinc zone.

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. Additional high-grade copper-zinc-germanium mineralisation also was discovered in the Fault Zone and in the Fault Zone Splay in the immediate footwall of the Fault Zone.

1.4 Sample Preparation and Analysis

1.4.1 Gécamines Sample Preparation and Analysis

Historical sampling and assaying was 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. Sulphur analysis was carried out by the 'classical' gravimetric method.

No data are available for QAQC protocols implemented for the Gécamines samples and therefore the Gécamines sample assays were considered to be less reliable than the KICO sample assays.

1.4.1.1 Resampling Programme

A comprehensive resampling programme was undertaken on historical Gécamines drill core from the Big Zinc zone and Fault Zone below 1,270 mRL 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 zone 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 QAQC 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 20 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 be 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.

1.4.1.2 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 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 water immersion (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 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.

1.4.2 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 Zn-Cu 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.

Sample preparation was completed by staff from KICO and its affiliated companies at its own internal containerised laboratories at Kolwezi and Kamoa. 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 by staff from that project. Representative subsamples were air freighted to BVM 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 g to 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; and
- 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, and
- Resampling of selected Gécamines drillholes which intersected the Fault Zone and Big Zinc zone.

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 sulphur 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 quality assurance and quality control (QAQC) programme was maintained by KICO throughout the underground drilling campaign comprising drillholes KPU001 to KPU097. The QAQC 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. QAQC data were exported from the MS Access database into software applications for creating monitoring charts and comparison charts.

The results of the QAQC 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.

1.5 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. The Central African Copperbelt lies within the Lufilian Arc, which comprises a 5–10 km thick sequence of metasedimentary rocks forming the Katanga Supergroup. These rocks were incorporated into a thin-skinned fold and thrust belt which resulted from the convergence of the Congo and Kalahari cratons. In the DRC, the Katangan Supergroup is defined by the Roan, Nguba and Kundulungu Groups.

The Kipushi Zn-Cu 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 mineralization hosted in pipe-like fault breccia zones, as well as tabular zones.

Mineralization 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 growth fault which was reactivated during the Lufilian Orogeny. Mineralization occurs in several distinct settings known as the Kipushi Fault Zone (copper, zinc and mixed copper-zinc mineralization both as massive sulphides and as veins), the Copper Nord Riche zone (mainly copper but also mixed copper-zinc mineralization, both massive and vein-style), the Série Récurrente zone (disseminated to veinlet-style copper mineralization), and the Big Zinc zone (massive zinc with local copper mineralization).

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

1.6 Metallurgical Testwork Summary

In 2013, approximately 60 kg of Kipushi quarter-core was delivered to Mintek, South Africa, for metallurgical testwork including; mineralogy, comminution and flotation testing. The composite sample head analysis was 38% Zn, 0.78% Pb, 0.4% Cu, 34% S, and 12% Fe. Mineralogy of the sample showed, as expected, sphalerite being predominate, 65.9%, followed by pyrite, 24%, with galena and chalcopyrite present in minor quantities. The major gangue was silica and carbonaceous minerals. The sphalerite and galena are coarse grained, grains up to 1 mm and 0.5 mm respectively. Chalcopyrite showed relatively fine grains, less than 0.04 mm.

Comminution testing showed the testwork sample to be soft, with Bond Ball Work Index of 7.8 kWh/t and SAG Milling Comminution (SMC) parameters A x b of 105.

Preliminary flotation tests indicated a zinc rougher recovery of 87% at 56% concentrate grade with a 50% passing 75 µm grind.

Although preliminary metallurgical testwork was encouraging, further testing was not undertaken whilst awaiting fresh samples from exploration drilling.

A second metallurgical sampling and testwork campaign was conducted in line with Kipushi resource development in early 2015; the Big Zinc zone was the primary focus of this programme. Six drillholes intercepting the Big Zinc zone were selected and intervals composited for metallurgical and mineralogical investigations. The samples came from hole numbers; KPU001, KPU003, KPU042, KPU051, KPU058, and KPU066. The drill core for the composite was selected to represent all mineralisation types in the Big Zinc zone including, but not limited to, Massive Brown Sphalerite (MSB), Massive Sulphide Mixed (MSM), and Dolomite (SDO). The target head grade for the composite sample was 37% Zn, based on the assayed intervals of the resource drill core. The head assay of the composite is presented in Table 1.5.

Table 1.5 Kipushi Composite Sample Head Analysis Results

Element	Zn (%)	Pb (%)	Fe (%)	Ca (%)	Si (%)	Cu (%)	Mg (%)	S (%)
Average head assay	40.10	1.45	5.97	6.20	1.73	0.27	3.55	25.45

Mineralogical investigations conducted on this composite head sample identified the main economic minerals in their order of abundance to be: sphalerite (67%), galena (2%), and chalcopyrite (1%); the main gangue minerals in the sample are dolomite (18%), followed by pyrite (8%) and quartz (3%).

Dense media separation (DMS) washability profiles were evaluated in the laboratory at three feed crush sizes using a combination of heavy liquid separation (HLS) and shaking tables. Fine material (–1 mm), mainly generated during crushing, was screened off ahead of HLS separation and tested on bench scale shaking tables (shaking tables provide a laboratory scale simulation of a commercial spiral plant). Fine material of –1 mm is not suitable for treatment by HLS.

The three crush sizes evaluated were namely: –20 mm, –12 mm, and –6 mm. Performance across the HLS and the shaking table, as a function of feed, was the same for all three crush sizes. The HLS circuit achieved 99% recovery at a concentrate grade of approximately 55% zinc; while the shaking table achieved 58% recovery at a concentrate grade of approximately 56% zinc. The difference in overall performance of the three crush sizes is the mass percentage reporting to the –1 mm fine fraction processed through the less-efficient shaking tables, this makes the results from the –20 mm sample superior because only 10% of feed bypasses the HLS compared to 22% and 32% of the –12 mm and –6 mm samples respectively. The –20 mm crush size achieved overall recovery of 95.4% at a saleable concentrate grade of 55.5% zinc.

In a commercial operation, ROM material will be crushed to produce a particle size of 100% passing –20 mm. This material will be screened at 1 mm, screen oversize material (–20+1 mm) will be concentrated through HLS and cyclones at a density of 3.1 g/cm^3 and the screen undersize material (–1 mm) will be upgraded through a spiral concentrator.

1.7 Mineral Resource Estimates

The Mineral Resource estimate was based on geochemical analyses and density measurements obtained from diamond drillhole core, which were completed by KICO between March 2014 and November 2015, with the cut-off date for data included in this estimate being 16 December 2015. 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. 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. Using the data from the drillholes, a three dimensional block model was created and the metal grades and density were estimated using ordinary kriging.

The Mineral Resource estimate was based on the results of 84 drillholes completed by KICO. Thirteen of the 97 holes drilled by KICO did not intersect the modelled zones. Minor amounts of mineralisation were sampled in nine of these 13 holes, the other four not intersecting any mineralisation of interest. An additional 107 historical holes drilled by Gécamines were used in the estimate.

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 1.6 for the predominantly zinc-rich bodies and in Table 1.7 for the predominantly copper-rich bodies.

The Mineral Resource estimate reported as at 23 January 2016 is the first Mineral Resource for Kipushi reported in accordance with CIM.

The Mineral Resources were categorized either as zinc-rich resources or copper-rich resources, depending on the most abundant metal. For the zinc-rich, Big Zinc and Southern Zinc, zones the Mineral Resource is reported at a base case cut-off grade of 7.0% Zn in Table 1.6, and the copper-rich, Fault Zone, Fault Zone Splay and Série Récurrente, zones at a base case cut-off grade of 1.5% Cu in Table 1.7.

Given the considerable revenue which will be obtained from the additional metals in each zone, MSA considers that mineralization 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, the level of study so far carried out being insufficient with which to do so.

Table 1.6 Kipushi Zinc-Rich Mineral Resource at 7% Zn cut-off grade, 23 January 2016

Zone	Category	Tonnes (millions)	Zn (%)	Cu (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
Big Zinc	Measured	3.59	38.39	0.67	0.36	18	17	54
	Indicated	6.60	32.99	0.63	1.29	20	14	50
	Inferred	0.98	36.96	0.79	0.14	7	16	62
Southern Zinc	Indicated	0.00	–	–	–	–	–	–
	Inferred	0.89	18.70	1.61	1.70	13	15	43
Total	**Measured**	**3.59**	**38.39**	**0.67**	**0.36**	**18**	**17**	**54**
	Indicated	**6.60**	**32.99**	**0.63**	**1.29**	**20**	**14**	**50**
	Measured & Indicated	**10.18**	**34.89**	**0.65**	**0.96**	**19**	**15**	**51**
	Inferred	**1.87**	**28.24**	**1.18**	**0.88**	**10**	**15**	**53**
Contained Metal Quantities								
Zone	**Category**	**Tonnes (millions)**	**Zn Pounds (millions)**	**Cu Pounds (millions)**	**Pb Pounds (millions)**	**Ag Ounces (millions)**	**Co Pounds (millions)**	**Ge Ounces (millions)**
Big Zinc	Measured	3.59	3,035.8	53.1	28.7	2.08	0.13	6.18
	Indicated	6.60	4,797.4	91.9	187.7	4.15	0.20	10.54
	Inferred	0.98	797.2	17.1	3.0	0.23	0.03	1.96
Southern Zinc	Indicated	0.00	0.0	0.0	0.0	0.00	0.00	0.00
	Inferred	0.89	368.6	31.8	33.5	0.38	0.03	1.23
Total	**Measured**	**3.59**	**3,035.8**	**53.1**	**28.7**	**2.08**	**0.13**	**6.18**
	Indicated	**6.60**	**4,797.4**	**91.9**	**187.7**	**4.15**	**0.20**	**10.54**
	Measured & Indicated	**10.18**	**7,833.3**	**144.9**	**216.4**	**6.22**	**0.33**	**16.71**
	Inferred	**1.87**	**1,168.7**	**49.6**	**36.8**	**0.61**	**0.06**	**3.21**

Notes:

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.
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 $1.02/lb, mining cost of $50/tonne, processing cost of $10 /tonne, G&A and holding cost of $10/tonne, transport of 55% Zn concentrate at $375/tonne, 90% zinc recovery and 85% payable zinc.

Table 1.7 Kipushi Copper-Rich Mineral Resource at 1.5% Cu cut-off grade, 23 January 2016

Zone	Category	Tonnes (millions)	Cu (%)	Zn (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
Fault Zone	Measured	0.14	2.78	1.25	0.05	19	107	20
	Indicated	1.01	4.17	2.64	0.09	23	216	20
	Inferred	0.94	2.94	5.81	0.18	22	112	26
Série Récurrenté	Indicated	0.48	4.01	3.82	0.02	21	56	6
	Inferred	0.34	2.57	1.02	0.06	8	29	1
Fault Zone Splay	Inferred	0.35	4.99	15.81	0.005	20	127	81
Total	**Measured**	**0.14**	**2.78**	**1.25**	**0.05**	**19**	**107**	**20**
	Indicated	**1.49**	**4.12**	**3.02**	**0.07**	**22**	**165**	**15**
	Measured & Indicated	**1.63**	**4.01**	**2.87**	**0.06**	**22**	**160**	**16**
	Inferred	**1.64**	**3.30**	**6.97**	**0.12**	**19**	**98**	**33**
Contained Metal Quantities								
Zone	**Category**	**Tonnes (millions)**	**Cu Pounds (millions)**	**Zn Pounds (millions)**	**Pb Pounds (millions)**	**Ag Ounces (millions)**	**Co Pounds (millions)**	**Ge Ounces (millions)**
Fault Zone	Measured	0.14	8.5	3.8	0.2	0.09	0.03	0.09
	Indicated	1.01	93.2	59.1	1.9	0.75	0.48	0.64
	Inferred	0.94	61.1	120.9	3.8	0.68	0.23	0.79
Série Récurrenté	Indicated	0.48	42.4	40.5	0.2	0.32	0.06	0.09
	Inferred	0.34	19.4	7.7	0.4	0.09	0.02	0.01
Fault Zone Splay	Inferred	0.35	38.9	123.3	0.0	0.23	0.10	0.92
Total	**Measured**	**0.14**	**8.5**	**3.8**	**0.2**	**0.09**	**0.03**	**0.09**
	Indicated	**1.49**	**135.7**	**99.6**	**2.1**	**1.08**	**0.54**	**0.73**
	Measured & Indicated	**1.63**	**144.1**	**103.4**	**2.3**	**1.16**	**0.58**	**0.82**
	Inferred	**1.64**	**119.4**	**251.8**	**4.3**	**1.00**	**0.35**	**1.73**

Notes:

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.
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 $2.97/lb, mining cost of $50/tonne, processing cost of $10/tonne, G&A and holding cost of $10/tonne, 90% copper recovery and 96% payable copper.

The Measured and Indicated Mineral Resource for the zinc-rich bodies has been tabulated using a number of cut-off grades as shown in Table 1.8, and the Inferred Mineral Resource in Table 1.9.

Table 1.8 Kipushi Zinc-Rich bodies Measured and Indicated Mineral Resource Grade Tonnage Table, 23 January 2016

Cut-Off (Zn%)	Tonnes (Millions)	Zn (%)	Contained Zn Pounds (Millions)	Cu (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
5	10.46	34.12	7,870.0	0.65	0.95	19	15	50
7	**10.18**	**34.89**	**7,833.3**	**0.65**	**0.96**	**19**	**15**	**51**
10	9.78	35.99	7,757.4	0.63	0.98	19	15	52
12	9.50	36.72	7,689.4	0.62	1.00	19	15	53
15	9.06	37.85	7,559.1	0.59	1.01	20	15	54

Notes:

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.
4. Metal quantities are reported in multiples of Troy Ounces or Avoirdupois Pounds.

Table 1.9 Kipushi Zinc-Rich bodies Inferred Mineral Resource Grade Tonnage Table, 23 January 2016

Cut-Off (Zn%)	Tonnes (Millions)	Zn (%)	Contained Zn Pounds (Millions)	Cu (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
5	1.89	27.98	1,168.8	1.19	0.88	10	15	53
7	**1.87**	**28.24**	**1,165.7**	**1.18**	**0.88**	**10**	**15**	**53**
10	1.82	28.85	1,154.8	1.17	0.88	10	15	54
12	1.75	29.47	1,139.8	1.15	0.87	10	15	55
15	1.56	31.42	1,082.1	1.08	0.83	10	15	57

Notes:

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.
4. Metal quantities are reported in multiples of Troy Ounces or Avoirdupois Pounds.

The Measured and Indicated Mineral Resource for the copper-rich bodies has been tabulated using a number of cut-off grades as shown in Table 1.10, and the Inferred Mineral Resource in Table 1.11.

Table 1.10 Kipushi Copper-Rich bodies Measured and Indicated Mineral Resource Grade Tonnage Table, 23 January 2016

Cut-Off (Cu%)	Tonnes (Millions)	Cu (%)	Contained Cu Pounds (Millions)	Zn (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
1.0	2.56	3.00	169.2	2.01	0.05	17	114	11
1.5	**1.63**	**4.01**	**144.1**	**2.87**	**0.06**	**22**	**160**	**16**
2.0	1.17	4.92	126.6	3.66	0.08	26	202	19
2.5	0.95	5.54	115.8	4.06	0.08	29	227	20
3.0	0.82	5.99	108.0	4.32	0.08	30	244	20

Notes:

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.
4. Metal quantities are reported in multiples of Troy Ounces or Avoirdupois Pounds.

Table 1.11 Kipushi Copper-Rich bodies Inferred Mineral Resource Grade Tonnage Table, 23 January 2016

Cut-Off (Cu%)	Tonnes (Millions)	Cu (%)	Contained Cu Pounds (Millions)	Zn (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
1.0	2.40	2.64	139.8	5.85	0.09	16	79	29
1.5	**1.64**	**3.30**	**119.4**	**6.97**	**0.12**	**19**	**98**	**33**
2.0	1.24	3.81	104.2	7.29	0.13	20	109	33
2.5	0.90	4.40	87.6	8.01	0.13	21	113	34
3.0	0.68	4.95	74.0	8.38	0.15	21	118	34

Notes:

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.
4. Metal quantities are reported in multiples of Troy Ounces or Avoirdupois Pounds.

Mineral Resource estimates were completed below the 1,150 mRL on the Big Zinc zone, Southern Zinc zone, Fault Zone and Série Récurrente zone, extensive mining having taken place in the levels above. Below 1,150 mRL, 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 mRL 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 27 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), is a member in good standing 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 mineralization and activity being undertaken as defined in National Instrument 43-101 Standards of Disclosure of Mineral Projects.

1.8 Mining

Historical mining at Kipushi was carried out from surface to approximately 1,220 m below surface (mRL) and occurred in three contiguous zones: the North and South zones of the Fault Zone, and the Série Récurrente zone 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 Zn-Cu Project, including a series of vertical mine shafts, with associated head frames, 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. It is expected to be recommissioned as 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 zone, through the 1,150 mRL haulage level. 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.

Figure 1.1 Schematic Section of Kipushi Mine

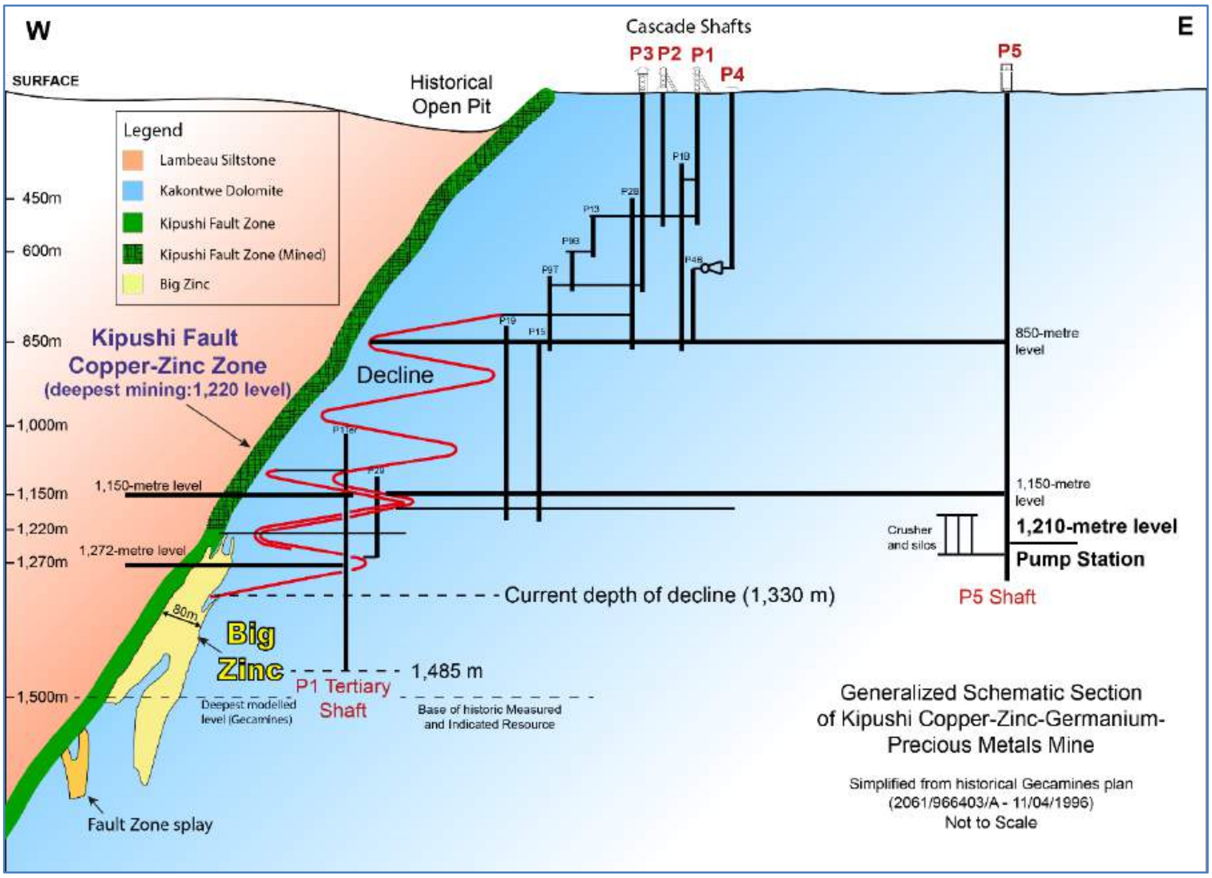


Figure by Ivanhoe, 2016.

The planned mining method is a combination of Sublevel Open Stoping (SLOS), Pillar Retreat, and Cut and Fill methods at a steady-state mining rate of 1.1 Mtpa. The existing and planned development and stoping is shown in Figure 1.2.

The primary mining method for the Big Zinc zone is expected to be SLOS, with cemented rock backfill. The crown pillars are expected to be mined once adjacent stopes are backfilled using the Pillar Retreat mining method. The Big Zinc zone is expected to be accessed via the existing decline and without significant new development. The main levels are planned to be at 60 m vertical intervals with sublevels at 30 m interval.

The Cut and Fill mining method has been identified to be used to extract the copper zone outside of the Big Zinc zone. In this method, mining occurs in horizontal slices, with the blasted copper material removed from the stopes, then crushed underground and sold at the mine gate.

Figure 1.2 Planned and Existing Development at Kipushi

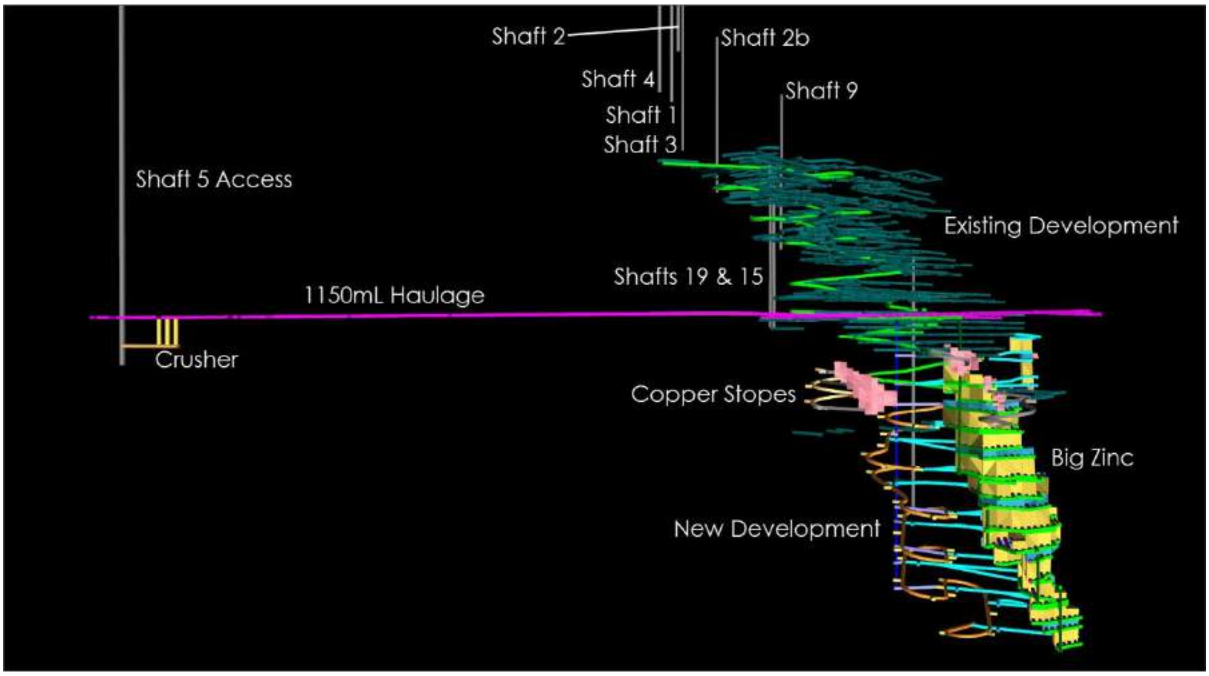


Figure by OreWin, 2016.

1.9 Processing Dense Media Separation

The planned process plant is a dense media separation (DMS) plant, which is expected to include crushing, screening, heavy liquid separation (HLS) and spirals to produce a high grade zinc concentrate. DMS is a simple density concentration technique that preliminary testwork has shown yields positive results for the Kipushi material, which has a sufficient density differential between the gangue (predominantly dolomite) and mineralisation (sphalerite). DMS washability profiles were evaluated in the laboratory at three feed crush sizes using a combination of HLS and shaking tables.

Preliminary test work results on three crush sizes indicated that –20 mm crush size resulted in the highest recovery and concentrate grade. This crush size achieved an overall recovery of 95.4% at a concentrate grade of 55.5% Zn.

The overall proposed process plant flowsheet (block flow diagram) is shown in Figure 1.3.

Figure 1.3 Overall Proposed Plant Flow Sheet

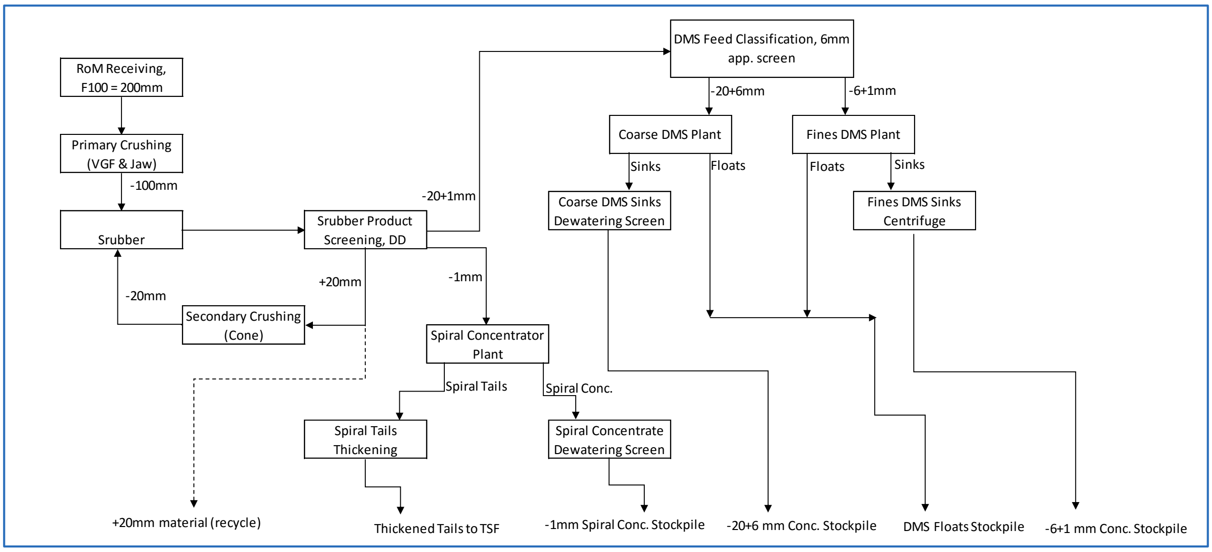


Figure by OreWin, 2016.

1.10 Infrastructure

The Kipushi Zn-Cu Project includes surface mining and processing infrastructure, concentrator, offices, workshops, and a connection to the national power grid. 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.

The surface infrastructure is owned by Gécamines, KICO has entered into an agreement to use the surface rights on the Kipushi Zn-Cu Project to the extent required for its operations.

An abundant supply of process water from the underground dewatering operations is expected to provide adequate water for processing and mining operations.

The overall proposed site layout is shown in Figure 1.4.

Figure 1.4 Overall Proposed Site Layout

Figure by Ivanhoe, 2016.

The Kipushi Station and connecting rail line from Kipushi to Manama and through to the Zambian border at Ndola, are owned and operated by La Société Nationale des Chemins de Fer du Congo (SNCC).

The proposed export route is to utilise the SNCC network from Kipushi to Ndola, connecting to the North-South Rail Corridor from Ndola to Durban. The Kipushi to Manama branch line will require a significant refurbishment over 30 km (the required capital for which is expected to be repaid through the transport costs). The North–South Rail Corridor from Sakania to Durban via Zimbabwe is fully operational and has a capacity of 5 Mtpa. Ivanhoe is working with Grindrod Limited, of South Africa, a leading and experienced freight services, shipping and financial services logistics operator in Southern Africa, to advance discussions with SNCC regarding the concession from Kipushi to Manama.

1.11 Production

Future proposed mine production has been scheduled to maximise the mine output and meet the DMS plant capacity. The mining production forecasts are shown in Table 1.12. Mine, process and concentrate production are shown in Figure 1.5 to Figure 1.7.

Table 1.12 Mining Production Statistics

Description	Unit	Total LOM	5-Year Average	LOM Average
Zinc Feed - Tonnes Processed				
Quantity Zinc Tonnes Treated	kt	9,394	981	939
Zinc Feed grade	%	32.15	32.65	32.15
Zinc Recovery	%	92.94	93.14	92.94
Zinc Concentrate Produced	kt (dry)	5,296	562	530
Zinc Concentrate grade	%	53.00	53.00	53.00
Copper Feed - Tonnes Processed				
Quantity Copper Tonnes Treated	kt	547	88	55
Copper Feed grade	%	5.41	5.68	5.41
Copper Recovery	%	90.00	90.00	90.00
Copper Concentrate Produced	kt (dry)	106	18	11
Copper Concentrate grade	%	25.00	25.00	25.00
Metal Produced				
Zinc	kt	2,807	298	281
Copper	kt	27	4	3

Figure 1.5 Zinc and Copper Tonnes Mined

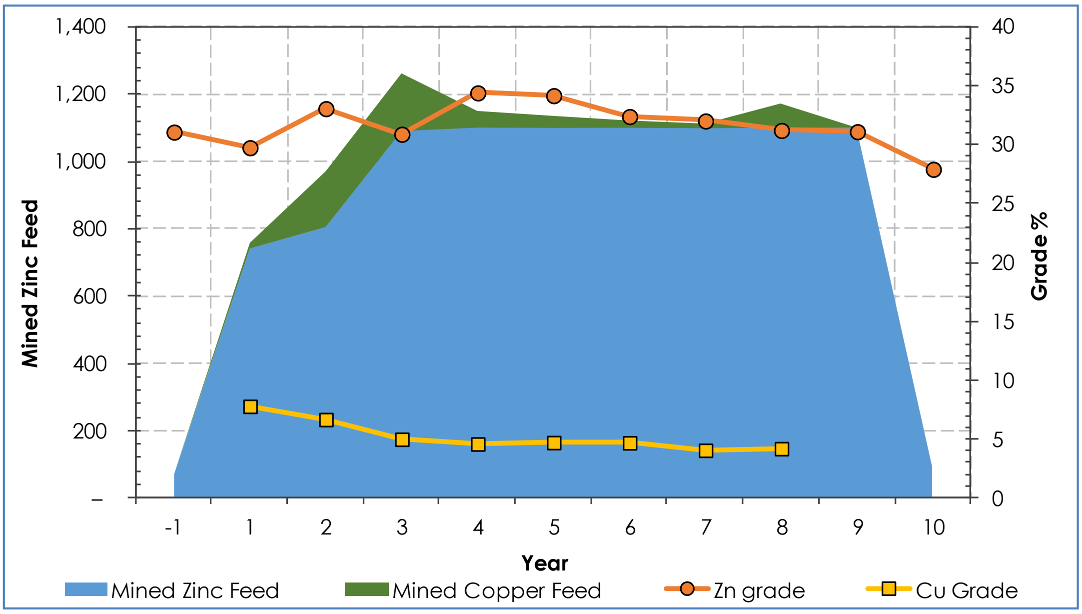


Figure 1.6 Zinc and Copper Tonnes Processed

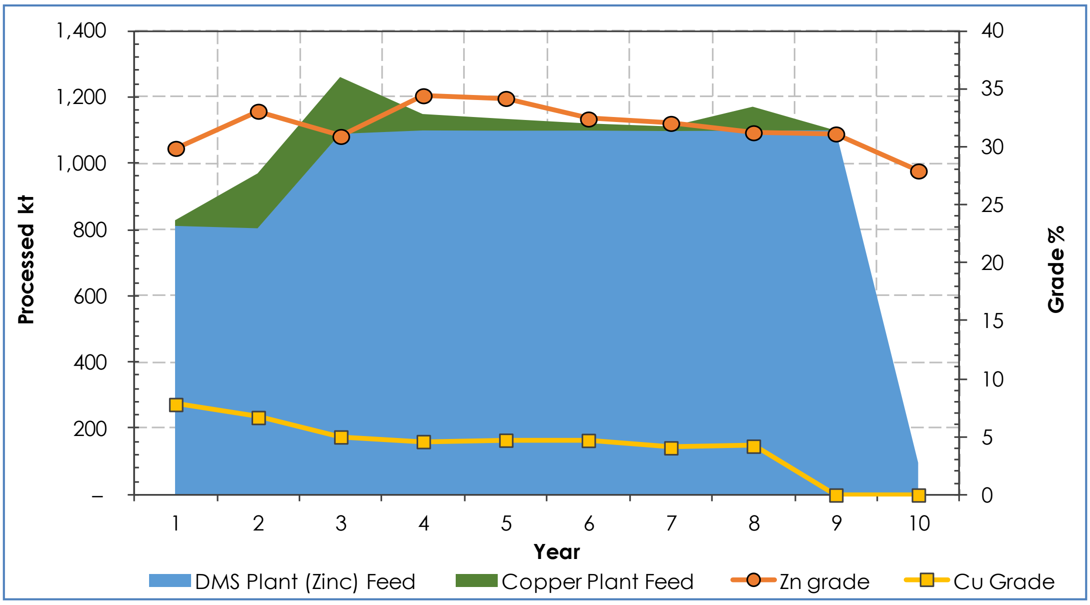
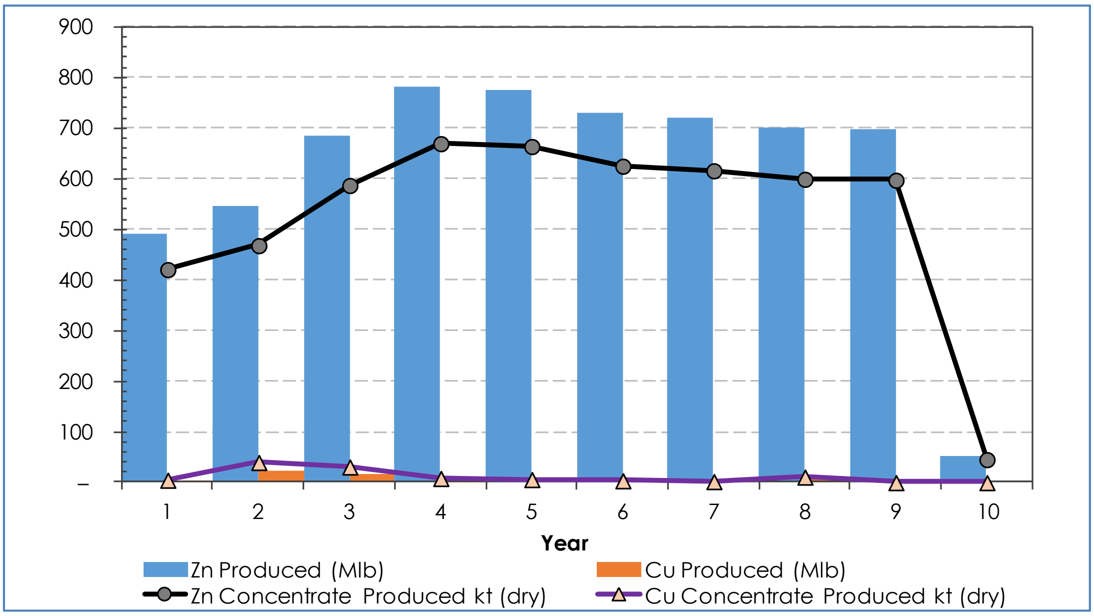


Figure 1.7 Concentrate and Metal Production



1.12 Economic Analysis

The estimates of cash flows have been prepared on a real basis as at 1 January 2016 and a mid-year discounting is used to calculate Net Present Value (NPV).

The projected financial results for undiscounted and discounted cash flows, at a range of discount rates, IRR and payback are shown in Table 1.13. The key economic assumptions for the discounted cash flow analyses are shown in Table 1.14. The results of NPV sensitivity analysis to a range of zinc prices and discount rates is shown in Table 1.15. A chart of the cumulative cash flow is shown in Figure 1.8.

Table 1.13 Financial Results

Description	Discount Rate	Before Taxation	After Taxation
Net Present Value ($M)	Undiscounted	1,473	1,076
	5.0%	973	696
	8.0%	759	533
	10.0%	642	444
	12.0%	542	368
Internal Rate of Return	–	36.4%	30.9%
Project Payback Period (Years)	–	2.1	2.2

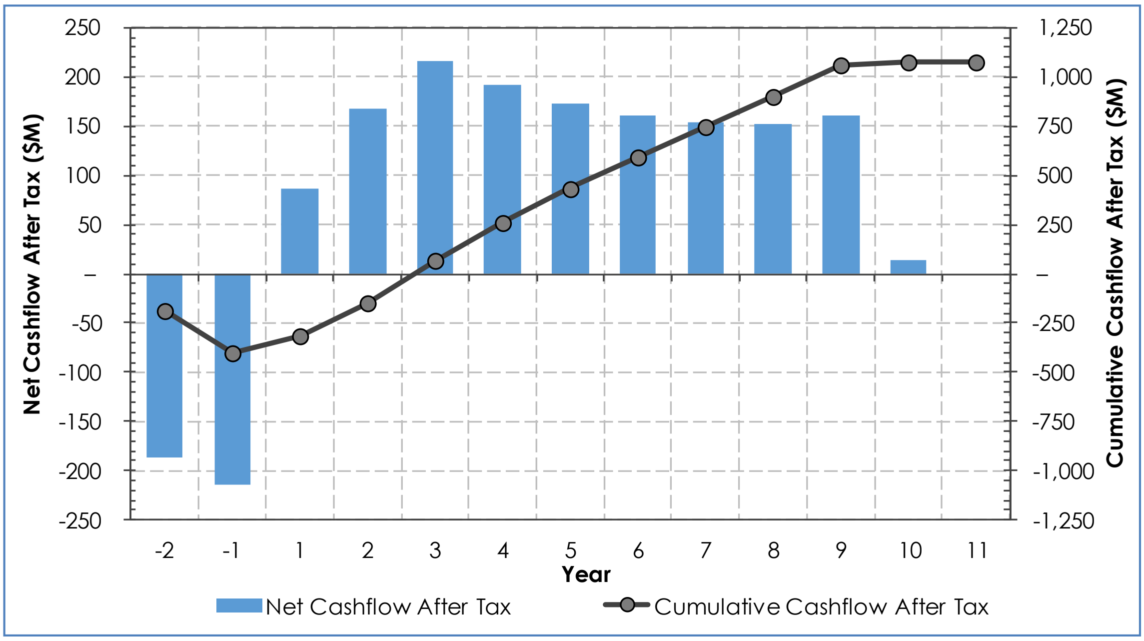
Table 1.14 Metal Prices and Terms

Parameter	Unit	Financial Analysis Assumption
Zinc Price	$/t	2,227
Copper Price	$/t	6,614
Zinc Treatment Charge	$/t concentrate	200.00
Copper Treatment Charge	$/t concentrate	90.00
Copper Refining Charge	$/t Cu	198.42

Table 1.15 After Tax Zinc Price Sensitivity – Discount Rates

Discount Rate	Zinc ($/t)						
	1,500	1,750	2,000	2,227	2,500	2,750	3,000
Undiscounted	-157	325	719	1,076	1,507	1,901	2,295
5%	-210	146	436	696	1,008	1,293	1,577
8%	-230	69	315	533	794	1,032	1,269
10%	-240	28	249	444	677	889	1,101
12%	-248	-7	193	368	577	767	957

Figure 1.8 Cumulative Cash Flow



The total capital cost estimates for the Kipushi Zn-Cu Project are shown in Table 1.16.

The estimated revenues and operating costs are presented in Table 1.17 along with the estimated net sales revenue value attributable to each key period of operation. The estimated cash costs are presented in Table 1.18.

Table 1.16 Estimated Capital Costs

Description	Pre-Production ($M)	Sustaining ($M)	Total ($M)
Mining:			
Rehabilitation	111	–	111
Underground	52	84	136
Capitalised Mining Operating Costs	37	–	37
Subtotal	**200**	**84**	**284**
Process & Infrastructure:			
Process & Infrastructure	32	6	38
Subtotal	**32**	**6**	**38**
Closure:			
Closure	–	20	20
Subtotal	**–**	**20**	**20**
Indirects:			
EPCM	29	2	32
Capitalised G&A & Other Costs	60	–	60
Subtotal	**89**	**2**	**92**
Others:			
Owners Cost	29	2	32
Capital Cost Before Contingency	**350**	**115**	**465**
Contingency	58	4	63
Capital Cost After Contingency	**409**	**119**	**528**

Table 1.17 Estimated Operating Costs and Revenues

Description	Total ($M)	5-Year Average	LOM Average
		($/t Milled)	
Revenue:			
Gross Sales Revenue	**5,481**	**555**	**551**
Less Realisation Costs:			
Transport Costs	1,466	147	147
Treatment & Refining Charges	1,074	108	108
Royalties	198	20	20
Total Realisation Costs	**2,737**	**275**	**275**
Net Sales Revenue	**2,744**	**279**	**276**
Less Site Operating Costs:			
Mining	536	58	54
Processing Zn and Cu (tolling)	87	10	9
General & Administration	120	11	12
Total	**743**	**79**	**75**
Operating Margin ($M)	**2,001**	**201**	**201**
Operating Margin (%)	**37**	**36**	**37**

Table 1.18 Estimated Cash Costs

Description	5-Year Average	LOM Average
	($/lb Zn)	
Mine Site Cash Cost	0.13	0.12
Realisation	0.45	0.44
Total Cash Costs Before Credits	**0.58**	**0.56**
Copper Credits	(0.04)	(0.03)
Total Cash Costs After Credits	**0.53**	**0.54**

1.12.1 Comparison to Other Projects

The Kipushi Project Mineral Resource Estimate, January 2016 includes Measured and Indicated Resources of 10.2 Mt at 34.89% 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 1.9).

Figure 1.9 Top 20 Zinc Projects by Contained Zinc

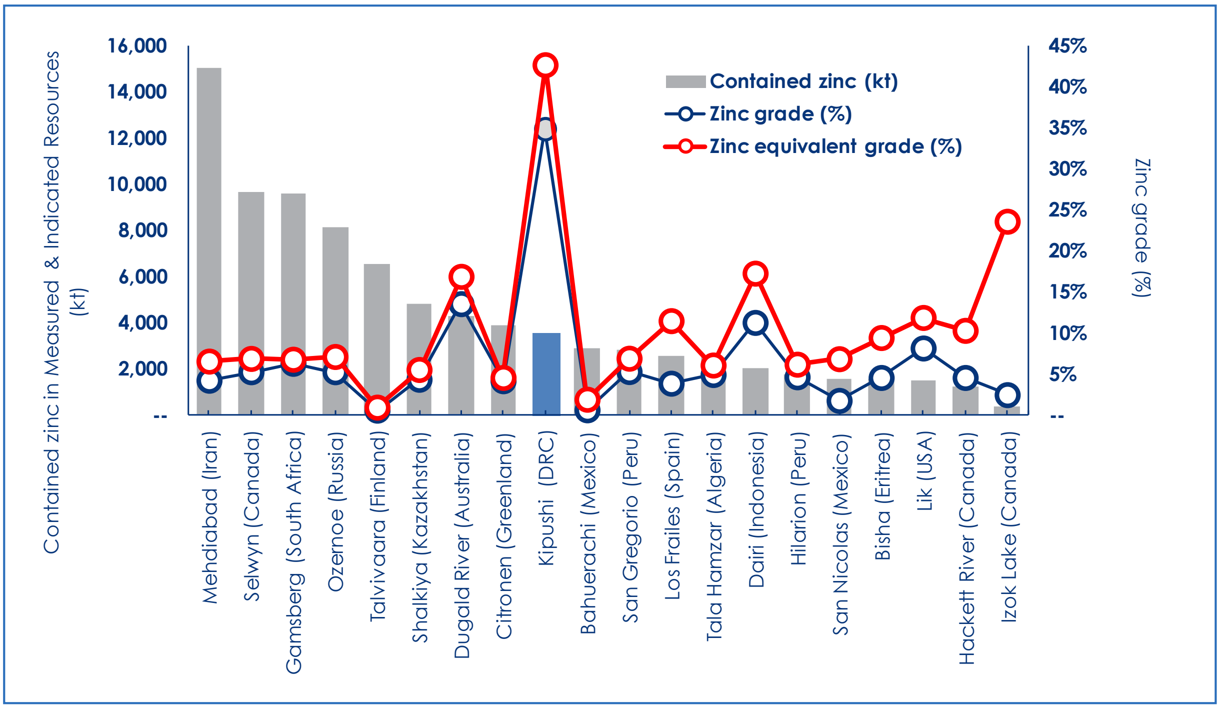


Figure by Wood Mackenzie, 2016.

Note: All tonnes and metal grades of individual metals used in the equivalency calculation of the above mentioned projects (except for Kipushi) are based on public disclosure and have been compiled by Wood Mackenzie. All metal grades have been converted by Wood Mackenzie to a zinc equivalent grade at price assumptions of US$1.01/lb Zn, US$2.86/lb Cu, US$0.91/lb Pb, US$12.37/lb Co, US$1,201/oz Au, US$17/oz Ag and US$2,000/kg Ge.

Life-of-mine average planned zinc concentrate production of 530,000 dry tpa, with a concentrate grade of 53% Zn, is expected to rank the Kipushi Zn-Cu Project, once in production, among the world's major zinc mines (Figure 1.10).

Figure 1.10 World's Major Zinc Mines ⁽¹⁾, Showing Estimated Annual Zinc Production and Zinc Head Grade

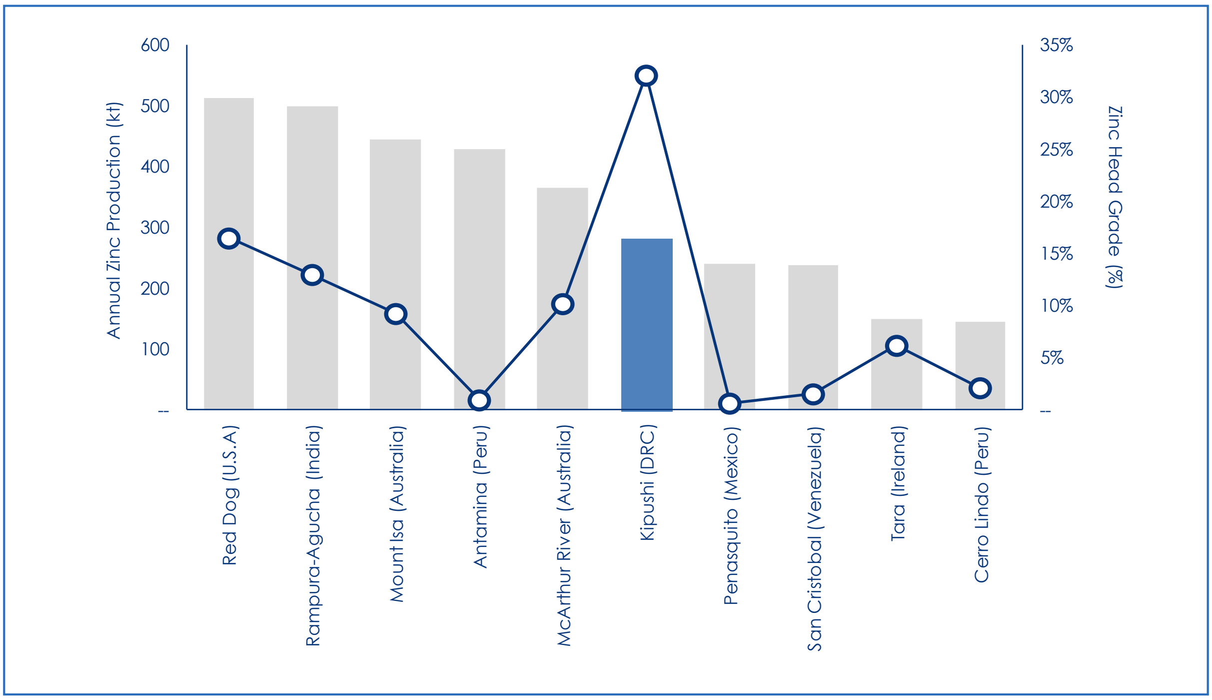


Figure by Wood Mackenzie, 2016.

(1) World's major zinc mines defined as the world's 10 largest zinc mines ranked by forecasted production by 2018.

Note: Independent research by Wood Mackenzie compared the Kipushi Project's life-of-mine average annual zinc production and zinc head grade of 281,000 tonnes and 32%, respectively, against production and zinc head grade forecasts for 2018.

Kipushi's estimated low capital intensity relative to comparable "probable" and "base case" zinc projects identified by Wood Mackenzie is highlighted in Figure 1.11.

Figure 1.11 Capital Intensity for Zinc Projects

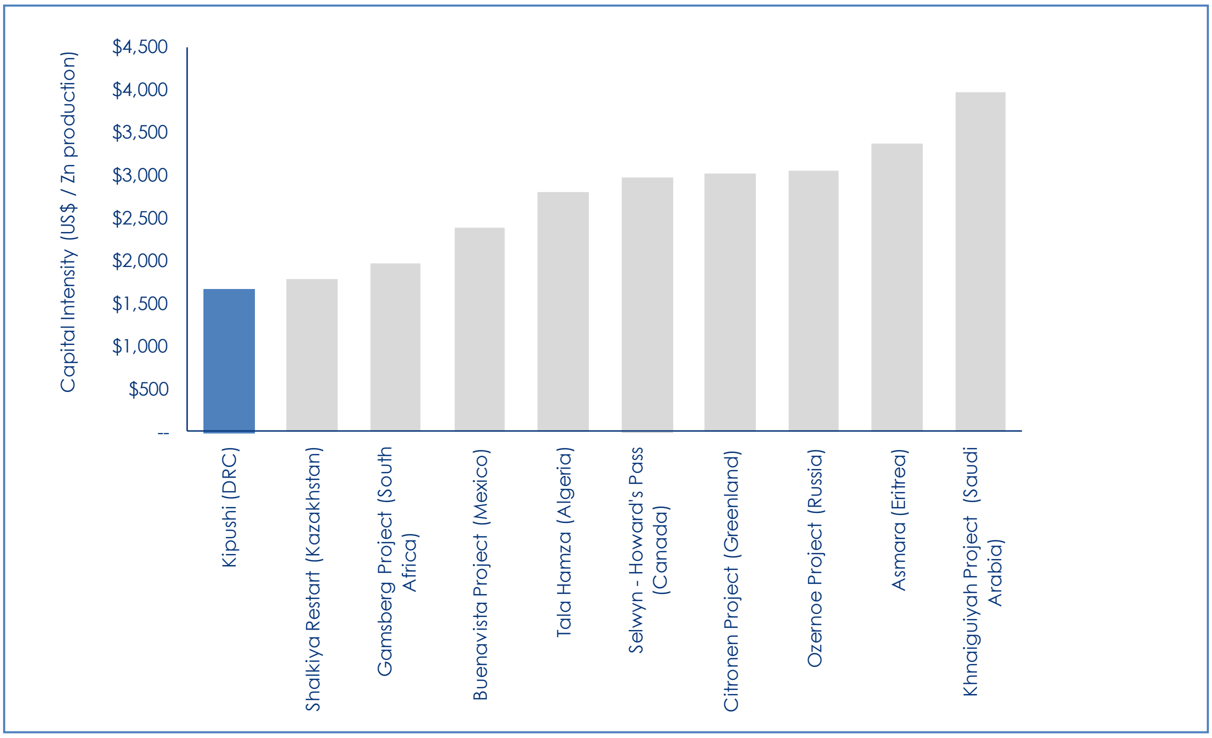


Figure by Wood Mackenzie, 2016.

Note: All comparable "probable" and "base case" projects as identified by Wood Mackenzie, based on public disclosure and information gathered in the process of routine research. The Kipushi 2016 PEA has not been reviewed by Wood Mackenzie.

Based on data from Wood MacKenzie, life-of-mine average cash cost of US$0.54/lb of zinc is expected to rank the Kipushi Zn-Cu Project, once in production, in the bottom quarter of the 2018 cash cost curve for zinc producers globally (Figure 1.12).

Figure 1.12 2018 Expected C1 Cash Costs

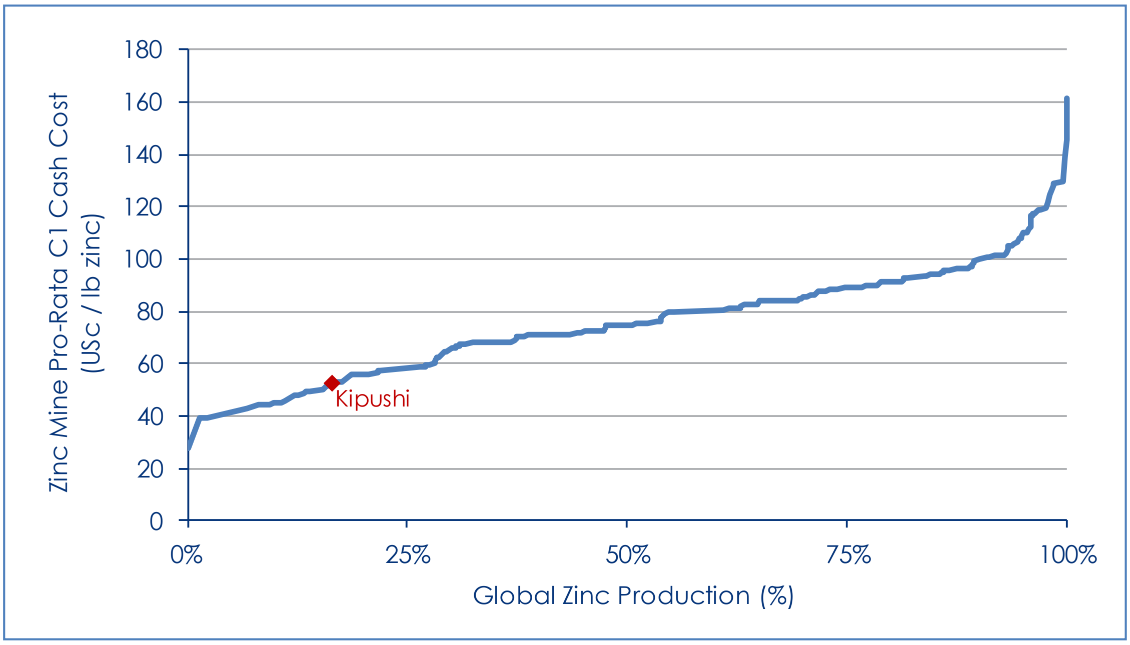


Figure by Wood Mackenzie, 2016.

Note: Represents C1 pro-rata cash costs which reflect the direct cash costs of producing paid metal incorporating mining, processing and offsite realization costs having made appropriate allowance for the co-product revenue streams. Based on public disclosure and information gathered in the process of routine research. The Kipushi 2016 PEA has not been reviewed by Wood Mackenzie.

1.13 Conclusions

The Kipushi 2016 PEA for the redevelopment of the Kipushi Mine is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorized as Mineral Reserves, and there is no certainty that the results will be realized. Mineral Resources do not have demonstrated economic viability and are not Mineral Reserves.

The Kipushi 2016 PEA has identified a positive business case and it is recommended that the Kipushi Zn-Cu Project is advanced to a pre-feasibility study level in order to increase the confidence of the estimates. There are a number of areas that need to be further examined and studied and arrangements that need to be put in place to advance the development of the Kipushi Zn-Cu Project.

The key areas for further work are:

Resources

- Further updates and resource estimation.
- Additional drilling of the lower Big Zinc zone and possible extensions.
- Test drilling of the copper zones.
- Additional resource estimation of other elements.

Geotechnical

- Further geotechnical drilling and logging will be required in the next stage of the project to increase the confidence in geotechnical data.
- The direction of drilling in the next stage should be along strike to avoid an orientation bias, as the majority of drilling at this stage is in the dip direction of the various mineralised zones.
- Laboratory testing of the rock units to investigate the rock properties of all rock units.
- Underground mapping should be carried out to improve confidence in the joint orientations and rock mass classification.

Mining

- Complete shaft and underground rehabilitation work.
- Additional study work to define the declines, ventilation, and material handling pass systems.
- Detailed design and optimisation including geotechnical recommendations.
- Prepare detail material flow designs.
- Mine stope and sequencing optimisation, and geotechnical review.
- Material handling / ventilation review and refinement of refurbishment requirements.

Process

- Further metallurgical testwork, aligned to predicted head grades, including DMS testwork on variability samples over a range of zinc feed grades and locations and bulk sample and pilot programme using DMS and spirals to confirm the design criteria across a DMS / Spiral circuit.
- Basic engineering for DMS and associated bagging plant.
- Copper rich-zone testwork.
- Study of the production of other metal concentrates or pure metals in particular copper, lead, cadmium, germanium, and silver.
- Study of the potential production of zinc calcine, zinc metal, and acid.

Infrastructure

- Define the rail option development.
- Progress agreements for rail transport and engage with the rail contractor.
- Evaluate container shipping with shipping companies.
- Investigate permitting of Kipushi station for the rail yard plans.
- Investigate the track conditions from Kipushi to the main Lubumbashi line.
- Containerisation.
- Analyse detailed power requirements and negotiate with supplier.
- Site survey.

Marketing

- Investigate customer uptake for container transport.
- Investigate copper sales at mine gate opportunities.

Environmental and Social

- Complete the regulatory Environmental Impact Statement (EIS) and the Environmental Management Plan (EMPP).
- Identify other permitting requirements.
- Prepare detailed closure plan.

Project Financing

- Investigate financing options and sources.
- Review of capital and operating cost estimates as part of the pre-feasibility study.

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 Project; (ii) the Platreef Project, and (iii) the Kipushi Zn-Cu Project. In addition, Ivanhoe holds interests in prospective mineral properties in the DRC and South Africa.

Kipushi Holding Limited (a subsidiary of Ivanhoe Mines Ltd. (Ivanhoe)) and La Générale des Carrières et Des Mines (Gécamines) have a joint venture agreement (JV Agreement) over the Kipushi Zn-Cu Project. Ivanhoe and Gécamines respectively own 68% and 32% of the Kipushi Zn-Cu Project through Kipushi Corporation SA (KICO), the mining rights holder of the Kipushi Zn-Cu 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.

2.2 Terms of Reference and Purpose of the Report

The Kipushi 2016 PEA is an Independent Technical Report on the Kipushi Zn-Cu Project prepared for Ivanhoe Mines Ltd. (Ivanhoe) as part of the strategy for redevelopment of the Kipushi Zn-Cu Project.

The Kipushi 2016 PEA is a Preliminary Economic Assessment with an effective date of 12 May 2016 that has been prepared using the June 2011 edition of Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects.

The following companies have undertaken work in preparation of the Kipushi 2016 PEA:

- OreWin: Overall report preparation, underground mining, mine geotechnical, mineral processing, infrastructure, and financial model.
- MSA: Geology, Drillhole data validation, Sample preparation, Analysis and Security, and Mineral Resource estimation.

This Report uses metric measurements. The currency used is US dollars (US$).

2.3 Principal Sources of Information

OreWin and MSA have based its review of the Project on information and data provided by KICO, along with other relevant published and unpublished data. The QPs have endeavoured, by making all reasonable enquiries, to confirm the authenticity and completeness of the technical data upon which the Technical Report is based.

Reports and documents listed in Section 3 and Section 27 of this Report were used to support preparation of the Report. Additional information was provided by Ivanhoe personnel as requested. Supplemental information was also provided to the QPs by third-party consultants retained by Ivanhoe in their areas of expertise.

2.4 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, 1.2, 1.6, 1.8 to 1.12, 1.13; Section 2; Section 3; Section 4 to 5; Section 13; Section 15 to 22; Section 23 to 24; Section 25; Section 26.1; Section 27.
- Michael Robertson, BSc Eng (Mining Geology), MSc (Structural Geology), Pr.Sci.Nat SACNASP, MGSSA, MSEG, MSAIMM, employed by MSA as a Principal Consulting Geologist was responsible for: Section 1.3 to 1.5, 1.13; Section 2; Section 3; Section 6 to Section 12; Section 25; Section 26.2; Section 27.
- Jeremy Witley, BSc Hons (Mining Geology), MSc (Eng), Pr.Sci.Nat SACNASP, FGSSA, employed by MSA as a Principal Resource Consultant was responsible for: Section 1.7, 1.13; Section 2; Section 3; Section 14; Section 25; Section 26.2; Section 27.

2.5 Site Visits and Scope of Personal Inspection

Site visits were performed as follows:

Mr Bernard Peters visited the Project from 1 June 2015 to 3 June 2015 and again on 11 September 2015. 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 February 2013 to 23 February 2013 and again from 22 April 2013 to 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 September 2014 to 11 September 2014 and again from 11 May 2015 to 13 May 2015.

2.6 Effective Dates

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

- Date of drillhole database close-out date for updated Mineral Resource estimate: 16 December 2015.
- Date of Mineral Resource update for mineralisation amenable to underground mining methods: 23 January 2016.

3 RELIANCE ON OTHER EXPERTS

The QPs, as authors of Kipushi 2016 PEA, 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 Zn-Cu Project area.

3.1 Mineral Tenure

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

- A copy of the exploitation permit ("Certificat d'Exploitation") PE12434 dated 22 July 2011, issued by Cadastre Minière (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 Gécamines and Kipushi Resources International Limited (KRIL). Ivanhoe purchased the original KRIL 68% interest in the project.

This Technical Report has been prepared on the assumption that the Kipushi Zn-Cu 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 program permitting status of the Kipushi Zn-Cu 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.

- Environmental Report on the Kipushi Zinc-Copper mine, Democratic Republic of Congo, 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.

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:

- 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 01 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.

This information was used in Sections 4 and 20 of the Report.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Kipushi Zn-Cu Project is located adjacent to the town of Kipushi in the south–eastern part of the Haut-Katanga Province in the DRC, adjacent to the border with Zambia (Figure 4.1). Kipushi town is situated approximately 30 km south–west of Lubumbashi, the capital of Haut-Katanga Province. The geographical location of the mine is 11°45’36” south and 27°14’13” east.

The Kipushi mine is a past-producing, high-grade underground zinc–copper mine in the Central African Copperbelt, which operated from 1924 to 1993. The mine produced approximately 60 Mt at 11.03% Zn and 6.78% Cu including, from 1956 through 1978, approximately 12,673 tonnes of lead and 278 tonnes 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 mRL. In 1993, the mine was put on care and maintenance due to a combination of economic and political factors.

Figure 4.1 Location of Kipushi near Lubumbashi in the DRC

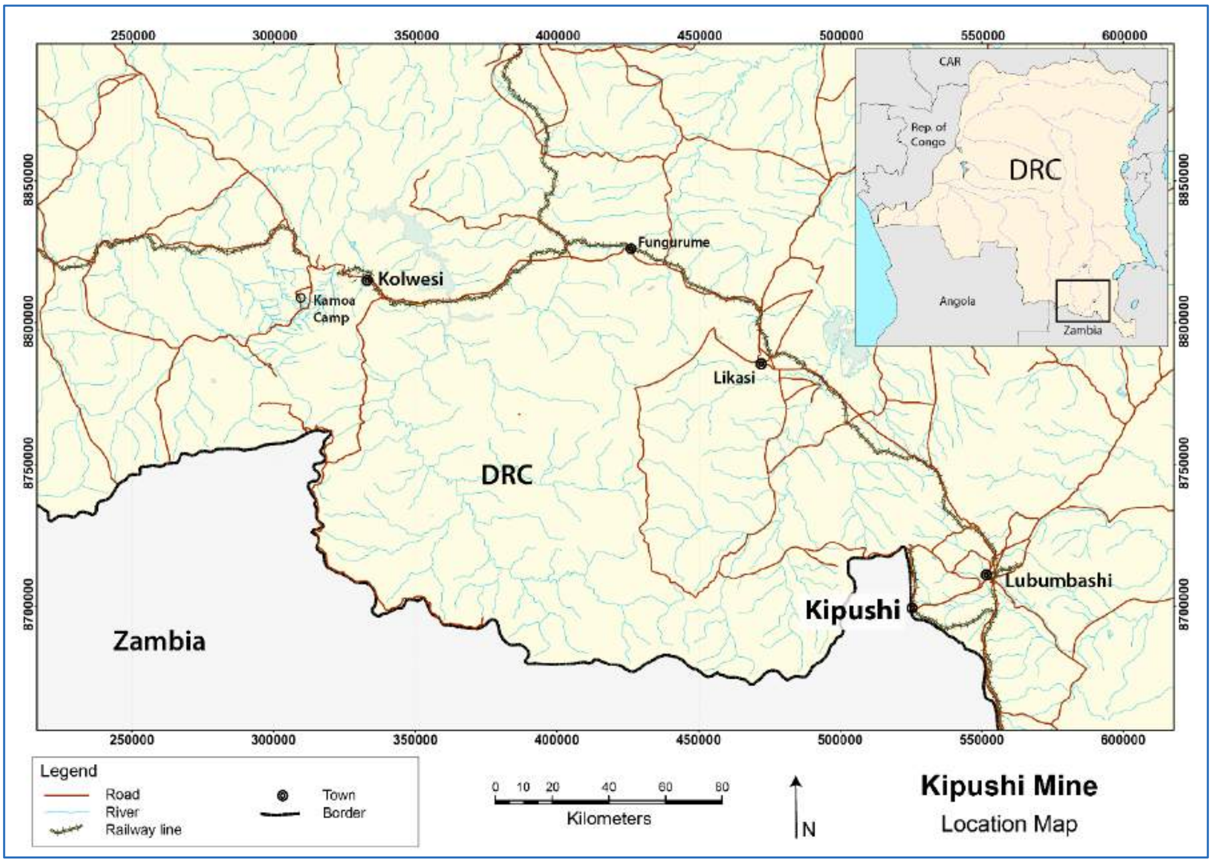


Figure by Ivanhoe, 2015.

4.2 Project Ownership

Ivanhoe Mines Ltd. (Ivanhoe) and La Générale des Carrières et Des Mines (Gécamines) have a joint venture agreement (JV Agreement) over the Kipushi Zn-Cu Project. Ivanhoe and Gécamines respectively own 68% and 32% of the Kipushi Zn-Cu Project through Kipushi Corporation SPRL (now Kipushi Corporation SA) (KICO), the mining rights holder of the Kipushi Zn-Cu 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. The JV Agreement was signed on 14 February 2007 and established KICO for the exploration, development, production and product marketing of the Kipushi Zn-Cu Project. The JV Agreement document is: Partnership Agreement No. 770/11068/SG/GC/2007 (including appendices 1 to 5, A to F, and later amendments 1 to 6) of 14 February 2007 between and Gécamines and Kipushi Resources International Limited (KRIL). Ivanhoe purchased the original KRIL 68% interest in the project.

4.3 Mineral Tenure

KICO holds the exclusive right to engage in mining activities within the Kipushi Zn-Cu Project area through a mining right, an exploitation permit (PE12434) valid until 3 April 2024 and covering 505 ha. This permit is renewable under the terms of the DRC Mining Code. The boundary coordinates of the permit area are shown in Table 4.1.

Exploitation permit (PE12434) granted KICO the right to mine and process copper and cobalt from the Kipushi Zn-Cu Project. On 15 June 2012, KICO submitted an application to the Cadastre Minière (CAMI), which resulted in the extension of the permit PE12434 for the extraction and processing of zinc, silver, lead, and germanium.

The Zambian and DRC governments have both contracted FelxiCadastra (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 PE12434 cross the international border, as do some of the co-ordinates 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 to mine stops at the international border, and any part of the exploitation permit area extending beyond the DRC borders are excluded from the licence.

The mineralisation at the Kipushi Zn-Cu 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 Zn-Cu Project only make reference to those Mineral Resources which lie within the DRC.

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

Permit Number	Type	Area (Ha)	Grant Date	Expiry Date	Point	Longitude			Latitude		
						Degree	Minute	Second	Degree	Minute	Second
PE12434	Exploitation Permit	505.0	2/7/2011	3/4/2024	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

* 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.

4.4 Surface Rights

KICO holds only the subsurface mineral title to the property, which includes ownership of the underground workings as well as the various mine shafts, headframes and related infrastructure. Appendix 5 of the JV Agreement details the assets including Shaft 5, the related infrastructure and underground workings that are made available by Gécamines for the purpose of redeveloping the Kipushi Zn-Cu Project.

Gécamines is the owner of the surface rights and surface infrastructure within the Kipushi Zn-Cu Project site, including but not limited to: (i) the older concentrator; (ii) the CMSK concentrator; (iii) the waste site; and (iv) the historical open-pit. The Kipushi Mine layout is shown in Figure 4.2.

There are a number of surface related activities occurring on the land which constitutes the Kipushi Zn-Cu Project in which KICO has no ownership or control. The CMSK Concentrator ceased operation in early 2015 with the sale of the Luiswishi mine by Gécamines, which supplied the feed to the plant.

KICO has the ability to utilise surface rights on the Kipushi Zn-Cu Project to the extent required in connection with mining operations.

Figure 4.2 Kipushi Existing Mine Layout

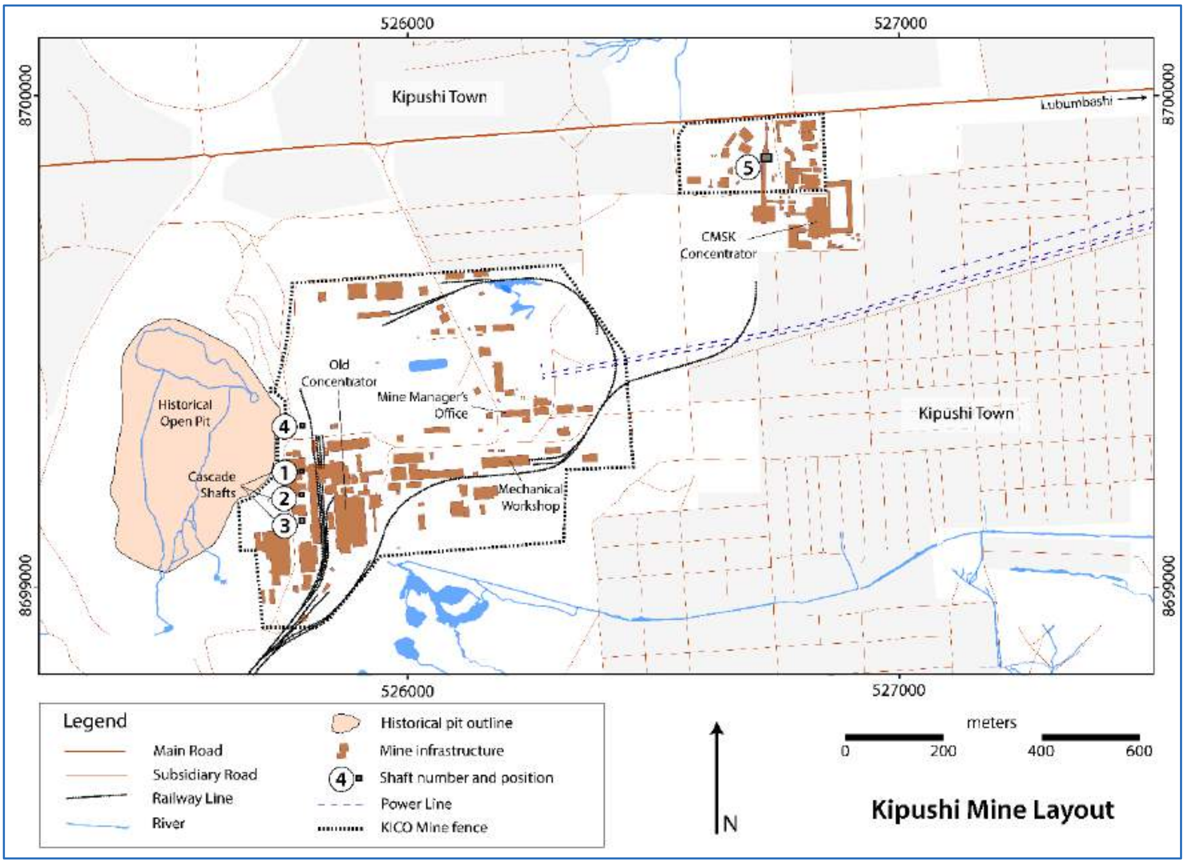


Figure by Ivanhoe, 2015.

4.5 Property Obligations and Agreements

A number of payments are required to keep the exploitation permit in good standing. Two fees levied annually are based on the number of cadastral squares held by permit type (surface rights fee) and on the surface area held under permits (land tax), as set out in the DRC 2002 Mining Code. As Exploitation Permit PE12434 is under Force Majeure, KICO will pay these fees only when the Force Majeure is over.

In addition, pursuant to the JV Agreement, KICO is required to pay to Gécamines a net turnover royalty of 2.5%, which, until the social loan (as defined in the JV Agreement) has been repaid in full (including accrued interest), is payable by way of offset against amounts owed by Gécamines under the social loan.

All payments relating to the current permits, licenses and agreements associated with the Kipushi Zn-Cu Project have been made and these permits, licenses, and agreements are held in good standing.

4.6 Environmental Liabilities

The property was the subject of an environmental audit by Direction in Charge of the Protection of the Mining Environment (DPEM) within the Ministry of Mines, in August 2011, who reported that all environmental obligations attached to the relevant licences had been discharged. 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 Ivanhoe acquired the Kipushi Zn-Cu Project in November 2011.

4.7 Significant Risk Factors

In February 2013, a draft law on the revision of the 2002 Mining Code was circulated by the DRC Minister of Mines. In February 2016 the DRC Minister of Mines announced that the current code will be retained for the foreseeable future.

4.8 Mining Legislation in the DRC

4.8.1 Mineral Property and Title

The following review of the DRC mineral legislation is summarised from Hubert André-Dumont, 2013, Kelly et al., 2012, and the 2002 Mining Code of the DRC.

The principal legislation governing mining activities in the DRC is the Mining Code (Law No. 007/2002) dated 11 July 2002 (the 2002 Mining Code). The applications of the Mining Code are provided by the Mining Regulations enacted by Decree No. 038/2003 of 26 March 2003 (the 2003 Mining Regulations). The legislation incorporates environmental requirements.

All mineral rights in the DRC are held by the State, and the holder of mining rights gains ownership of the mineral products for sale. Under the 2002 Mining Code, mining rights are regulated by Exploration Permits (Permis de Recherches or PR), Exploitation Permits (Permis d’Exploitation, or PE), small-scale Exploitation Permits and tailings Exploitation Permits (Permis d’Exploitation de Rejets, or PER).

Under the 2003 Mining Regulations, the DRC is divided into mining cadastral grids using a WGS84 geographic coordinate system. The grid defines uniform quadrangles or cadastral squares (carrés cadastraux) typically 84.955 ha in area. The perimeter of a mining right is in the form of a polygon consisting of entire contiguous quadrangles subject to the limits relating to the borders of the DRC and those relating to reserved prohibited and protected areas as set forth on the 2003 Mining Regulations. Perimeters are exclusive and may not overlap, except where the Mining Code & Regulations authorize overlapping.

4.8.2 Exploitation Permits

Exploitation permits are valid for 30 years and renewable for 15 year periods until the end of the mine's life, provided the conditions laid out in the 2002 Mining Code have been met. Granting of a permit is dependent on a number of factors that are defined in the 2002 Mining Code, including:

- Proof of the existence of a deposit which can be economically exploited, by presenting a feasibility study, accompanied by a technical framework plan for the development, construction, and exploitation of the mine.
- Proof of the existence of the financial resources required for the carrying out of the project, according to a financing plan for the development, construction and exploitation of the mine, as well as the rehabilitation plan for the site when the mine is closed. This plan specifies each type of financing, the sources of planned financing and justification of their possible availability.
- Pre-approval of the project's Environmental Impact Statement (EIS) and the Environmental Management Protection Plan (EMPP).
- Transfer to the DRC State 5% of the shares in the registered capital of the company applying for the licence. These shares are free of all charges and cannot be diluted.

As the right to mine already exists on the property and are derived from mining rights under prior legislation, these obligations are not applicable to the property.

The PE, as defined in the 2002 Mining Code, allows the holder the exclusive right to carry out, within the perimeter over which it has been granted, and during its term of validity, exploration, development, construction and exploitation works in connection with the mineral substances for which the licence has been granted, and associated substances if the holder has applied for an extension. So long as a perimeter is covered by an exploitation permit, no other application for a mining or quarry right can be granted within such perimeter. The holder of a PE has the right to extend its permit to include those minerals that it can demonstrate are associated minerals. Associated minerals are those in-situ minerals that are necessarily extracted simultaneously with the minerals listed in the original permit.

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

- Enter the exploitation perimeter to conduct mining operations.
- Build the installations and infrastructures required for mining exploitation.
- Use the water and wood within the mining perimeter for the requirements of the mining exploitation, complying with the requirements set forth in the EIS and the EMPP.

- Use, transport and freely sell the products originating from within the exploitation perimeter.
- 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.
- Proceed to carry out works to extend the mine.

A PE expires at the end of the appropriate term of validity if no renewal is applied for in accordance with the provisions of the 2002 Mining Code, or when the deposit that is being mined is exhausted.

4.8.3 Sale of Mining Products

Under the 2002 Mining Code, the sale of mining products which originate from the exploitation permit is free, meaning that the holder of a PE may sell any licensed products to a customer of choice, at prices freely negotiated. However, the authorisation of the appropriate DRC Minister is required under the 2002 Mining Code for exporting unprocessed ores for treatment outside the DRC. This authorisation will only be granted if the holder who is applying for it demonstrates at the same time:

- The fact that it is impossible to treat the substances in the DRC at a cost that is economically viable for the mining project.
- The advantages for the DRC if the export authorisation is granted.

4.8.4 Surface Rights Title

The DRC State has exclusive rights to all land, but 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 2002 Mining Code states that subject to any rights of third parties over the surface concerned, the holder of an exploitation mining right has, with the authorisation of the governor of the province concerned, and on the advice of the Administration of Mines, the right to occupy within a granted mining perimeter the land necessary for mining and associated industrial activities, including the construction of industrial plants and dwellings, water use, dig canals and channels, and establish means of communication and transport of any type.

Any occupation of land that deprives surface right holders from using the surface, or any modification rendering the land unfit for cultivation, entails an obligation on the part of the mining rights holder to pay fair compensation to the surface right holders. The mining rights holder is also liable for damage caused to the occupants of the land in connection with any mining activity, even if such an activity has been properly permitted and authorised.

4.8.5 Royalties

According to the 2002 Mining Code a company holding a PE is subject to mining royalties. The royalty is due upon the sale of the product and is calculated at 2% of the price 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. Different rates apply to different types of metals sold. The holder of the mining licence will benefit from a tax credit equal to one third of the mining royalties paid on products sold to an entity carrying out transformation of mineral substances located in the DRC. Mining royalties paid may be deducted for income tax purposes.

4.8.6 Environmental Obligations

The 2002 Mining Code contains environmental obligations that have to be met as part of the mining right application. These are the preparation of an EIS and an EMPP. The 2002 Mining Code provides for a biennial environmental audit. If a company does not pass this audit, it may lose its permit. Upon mine closure, shafts must be filled, covered or enclosed, and a certificate obtained confirming compliance with environmental obligations under the terms of the approved environmental impact study and environmental management plan.

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. Kipushi is connected to Lubumbashi by a paved road. The closest public airport to the Kipushi Zn-Cu Project is at Lubumbashi where there are daily domestic, regional, and international scheduled flights.

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 Zn-Cu Project operated year-round, and it is expected that any future mining activities at the Kipushi Zn-Cu Project would also be able to 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 m above mean sea level 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 Zn-Cu Project area and near the mine's infrastructure and underground access.

Although the town of Kipushi is theoretically administered independently of the mine, 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.

The mine was the largest employer of the local population prior to the suspension of mining operations in 1993. Since that time a number of mine personnel have been retained on the care-and-maintenance operations and to keep the mine secure. Many of these people still live in the area. As of 31 December 2014, KICO 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 Zn-Cu Project, including a series of vertical mine shafts, with associated head frames, to various depths, as well as underground mine excavations. The newest shaft (Shaft 5) is 8 m in diameter and 1,240 m deep with a lowest operating level at 1,150 mRL. It provides the primary access to the lower levels of the mine, including the Big Zinc zone. It has three independent friction hoists, and all compartments remain operational. The condition of the facility is fair, but will require a refurbishment program 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 mRL. This station failed in 2011 and water level rose to 862 mRL at its peak. Since Ivanhoe has assumed responsibility for ongoing rehabilitation and pumping, the water level has been lowered and stabilised at approximately 1,300 mRL on the Cascades Shaft 1 Tertiary (allowing underground diamond drilling from the 1,272 mRL hangingwall drive). The underground infrastructure, including the crushing system, which has been exposed since dewatering, is in relatively good order.

The Kipushi Zn-Cu Project includes surface mining and processing infrastructure, concentrator, offices, workshops, and a connection to the national power grid. 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. All of the surface infrastructure is owned by Gécamines.

The bulk of the Mineral Resources, and exploration potential, lie adjacent to or below the 1,150 mRL 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 completion of dewatering at the end of 2013. Hangingwall drill stations are present on 1,132 mRL and 1,272 mRL, and an underground decline is developed in the footwall to approximately 1,330 mRL. The re-establishment of operations at the Kipushi Zn-Cu Project would require refurbishment of underground access via Shaft 5, and construction of new processing and disposal facilities. Process water for any planned mining operation could be obtained from the underground pumping operations.

5.4 Surface Rights

Surface rights (which are distinct from mining rights) for the Kipushi Zn-Cu 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, including 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 pursuant to which KICO will be required to pay rental fees of $100,000 per month in exchange for the exclusive right to use the surface infrastructure held by Gécamines. 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.

6 HISTORY

6.1 Prior Ownership and Ownership Changes

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 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 mRL, 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, which resulted in the February 2007 Kipushi JV Agreement and the creation of the joint venture company, KICO. The Kipushi JV Agreement was novated to the Kipushi Vendor by United Resources AG via a novation act in May 2008 and Kipushi Vendor replaced 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.

The Big Zinc zone, adjacent to the Fault Zone on the footwall side, was discovered shortly before the mine suspended production, and has never been mined, although the currently decline extends to approximately 1,330 mRL. The mine flooded in early 2011 due to a lack of pumping maintenance over an extended period. After acquiring a 68% interest in Kipushi in November 2011, Ivanhoe assumed responsibility for ongoing rehabilitation and pumping. Gécamines hold the remaining 32% interest in Kipushi.

6.2 Historical Exploration

Between 1974 and 1993, Gécamines drilled a total of 762 holes between 850 mRL and 1,270 mRL for a total of 93,000 m (Kelly et al., 2012). Approximately 7,500 samples were submitted to the mine laboratory for routine analysis.

As at 1993, exploration drilling had traced the main Fault Zone to approximately 1,800 mRL.

The Big Zinc zone was investigated by diamond drilling carried out by Gécamines between 1990 and 1993. Resources below 1,150 mRL have been largely established through the drilling of about 200 diamond drillholes from two drill drives located in the hangingwall of the deposit at 1,132 mRL and 1,272 mRL. The Big Zinc zone was intersected by 84 of these holes. There has also been some underground sampling between 1,150 mRL and 1,295 mRL. Gécamines carried out all of this work prior to 1993. On 1,270 mRL, holes were drilled to intersect the Fault Zone and the Big Zinc zone on fans at 15 m spaced sections with holes inclined at between –25° and –90° (Figure 6.1).

Drill core is preserved from 49 of these holes and is stored on site at Kipushi. Most of the drill core is in reasonable condition as shown in Figure 6.2. In general, only mineralised intersections were retained by Gécamines, with only minor barren or sterile zones preserved in the core trays. The basis for defining sterile zones was a visual cut-off of 1% Cu and/or 7% Zn. The sterile zones are observed to contain variable sphalerite mineralisation in the form of veins and disseminations. Only minimal sterile material was available for resampling.

Four of the Gécamines holes drilled sub-parallel to mineralisation down to the 1,640 mRL enabled a hypothetical projection of the Big Zinc zone to the 1,800 mRL.

Figure 6.1 Extent of Gécamines Underground Drilling (below 1,042 mRL)

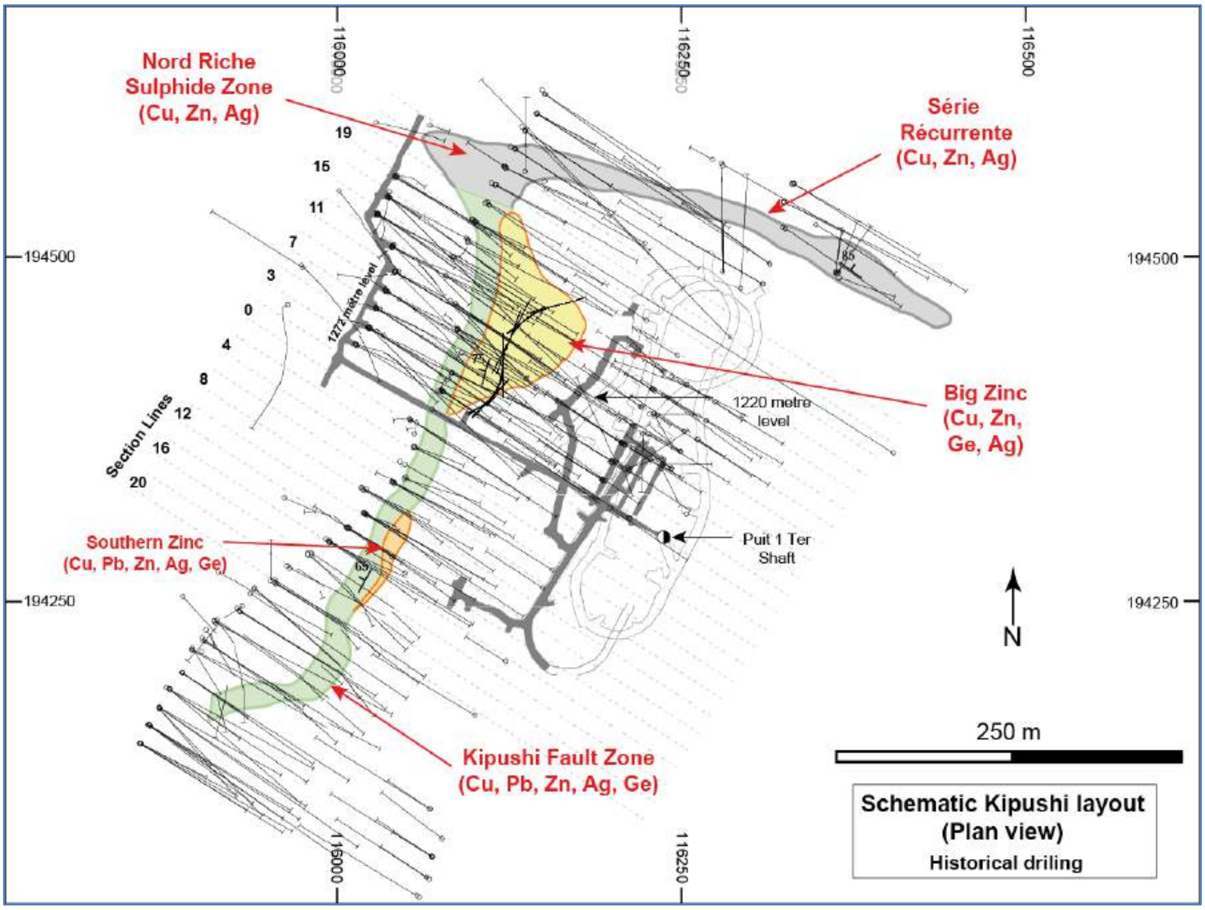


Figure by Ivanhoe, 2015.

Figure 6.2 Examples of the State of Preservation of Gécamines Drill Core

(a) base of the copper-rich Fault Zone and rare preservation of sterile dolomite in the footwall, and (b) massive zinc-rich mineralisation within the Big Zinc (drillhole 1270/9/V+30/-40/SE)

6.3 Historical Mineral Resource Estimates

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

Three historical resource estimates have been prepared on the Kipushi Zn-Cu 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.

In March 2016, Ivanhoe filed the Kipushi Project Mineral Resource Estimate, January 2016 dated 23 January 2016.

6.3.1 Gécamines Estimation Methodology

Gécamines adopted a classical estimation approach at Kipushi as described in Kelly et al., (2012). Underground drilling was initially carried out along 15 m spaced sections along drives developed parallel to the mineralised zone. Subsequently, sub-level cross-cuts were driven at 10 m intervals across the mineralised zone, allowing for detailed sampling of the zone. The drillhole and crosscut sampling were used to construct a series of 1:500 scale level plans spaced at 12.5 m vertical intervals, onto which grade categories were traced, using a minimum mining width of 5 m. The areas on the level plans were then projected halfway to the next level (6 m) for volume estimation and subsequent tonnage estimates using the regression formula:

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

Although assays were done for iron, there appears to have been no density factor (%Fe) generally applied for pyrite. Tonnage determinations may have underestimated high grade, low iron sphalerite mineralisation.

The Gécamines density factor was used mainly for mineralised zones other than the Big Zinc zone, as Gécamines was principally interested in copper. This density factor is therefore likely to be inappropriate for the estimation of zinc in high grade iron-poor sphalerite such as occurs in the Big Zinc zone. With the emphasis on copper, Gécamines adopted the following cut-off grade factors, based on 1970s metal prices:

- High grade: >2% Cu or >14% Zn
- Low grade: 1%–2% Cu or 7%–14% Zn
- Waste: <1% Cu or <7% Zn

By using this cut-off grade formula, material grading 2% Cu and 0% Zn would be considered for mining, whereas material grading 1.9% Cu and 13.9% Zn would not. Low-grade material, as defined above, was only mined when it occurred within a high grade intersection. The grade categories were outlined on level plans.

In order to validate the Gécamines density approach, KICO made 12 density determinations from a range of Big Zinc zone mineralisation styles, arriving at an average density of 3.85 g/cm^3.

The cut-off parameters were applied and resources/reserves classified as Certain, Probable, and Possible. The Certain category was supported by the results of detailed sampling in cross-cuts as well as from drillholes. The Probable category was based on a reasonable number of drillhole intersections and the assumption of continuity between them. Possible resources were based on the results of a few drillhole intersections and the projection of known geological controls on mineralisation. No allowance was made in these estimates for dilution or mining recovery; instead a mine call factor was applied to estimate the actual recovery.

6.3.2 Historical Estimate

Techpro collated the drillhole data, with the results being encoded by a local DRC team. This database incorporated the information contained in the drill log sheets as follows: (i) drillhole number; (ii) collar position, direction (azimuth), inclination, length, core recovery, date of completion, remarks; (iii) assay results for arsenic, copper, lead, zinc, sulphur, and iron; (iv) geological log, by means of simple codes; (v) mineralogical log, by means of codes; (vi) downhole survey data; and (vii) hydrological data. The Techpro established database, which includes data from 762 holes drilled at the Kipushi deposit, showed that the average length of all holes was 122 m with an average core recovery of 84%. Of these approximately 200 holes were drilled at or below 1,150 mRL and had an average drillhole length of 160 m and core recovery of 89%. Mineralisation, believed to form part of the Big Zinc zone, was intersected by 84 of these holes. The average length of all core samples sent for analysis (nearly 7,500 samples) was 3.44 m.

Some mineralisation extends into neighbouring Zambia, however this is not included in the historical estimate (Figure 6.3).

The Gécamines cut-off grade criteria were used in the Techpro estimate. The estimate is based on the Gécamines information and in particular the level plans. Where possible, Techpro checked the Gécamines figures and concluded that they were mostly acceptable and representative of the deposit. The Gécamines categories Certain, Probable, and Possible were considered by Techpro to be closely equivalent to the respective JORC categories of Measured, Indicated, and Inferred, and therefore applied these classifications.

The resources stated in Table 6.1 include the Copper Nord Riche, Fault Zone, and Big Zinc mineralised zones. Measured and Indicated Mineral Resources for the Big Zinc zone extend from 1,207 mRL to 1,500 mRL and total 4.71 Mt at an average grade of 38.55% Zn. Gécamines discovered the Big Zinc zone of mineralisation prior to placing the mine on care-and-maintenance in 1993. This previously unmined zone occurs between 1,200 mRL and 1,550 mRL with approximate dimensions of 100 m strike length by 40–80 m width by greater than 300 m plunge length.

Table 6.1 Techpro 1997 Historical Estimate

Resource Category	Tonnes	% Copper	% Zinc
Measured	8,899,979	2.53	9.99
Indicated	8,029,127	2.09	24.21
Total Measured and Indicated	16,929,106	2.32	16.76
Inferred to 1,800 mRL	9,046,352	1.93	23.32
Totals shown above include the following for the Big Zinc zone:			
Measured	793,086	1.16	33.52
Indicated	3,918,366	0.68	39.57
Measured & Indicated	**4,711,452**	**0.76**	**38.55**

Notes:

The above estimate is based on Gécamines information including the Gécamines cut-off grade approach.

Historical resource estimates presented are inclusive of the historical resource estimates attributable to the Big Zinc zone.

Source: Kelly et al., 2012

The reader is cautioned that a Qualified Person has not done sufficient work to classify the Historical Estimate as current Mineral Resources and the issuer is not treating the Historical Estimate as current Mineral Resources. The Historical Estimate should be regarded as no longer relevant, it having been superseded by the 23 January 2016 Mineral Resource. The Historical Estimate was prepared by Techpro in accordance with the 1996 edition of the JORC Code but would not meet current JORC or CIM standards.

Figure 6.3 Extent of Kipushi Mineralisation at the 1,150 mRL as at 1993

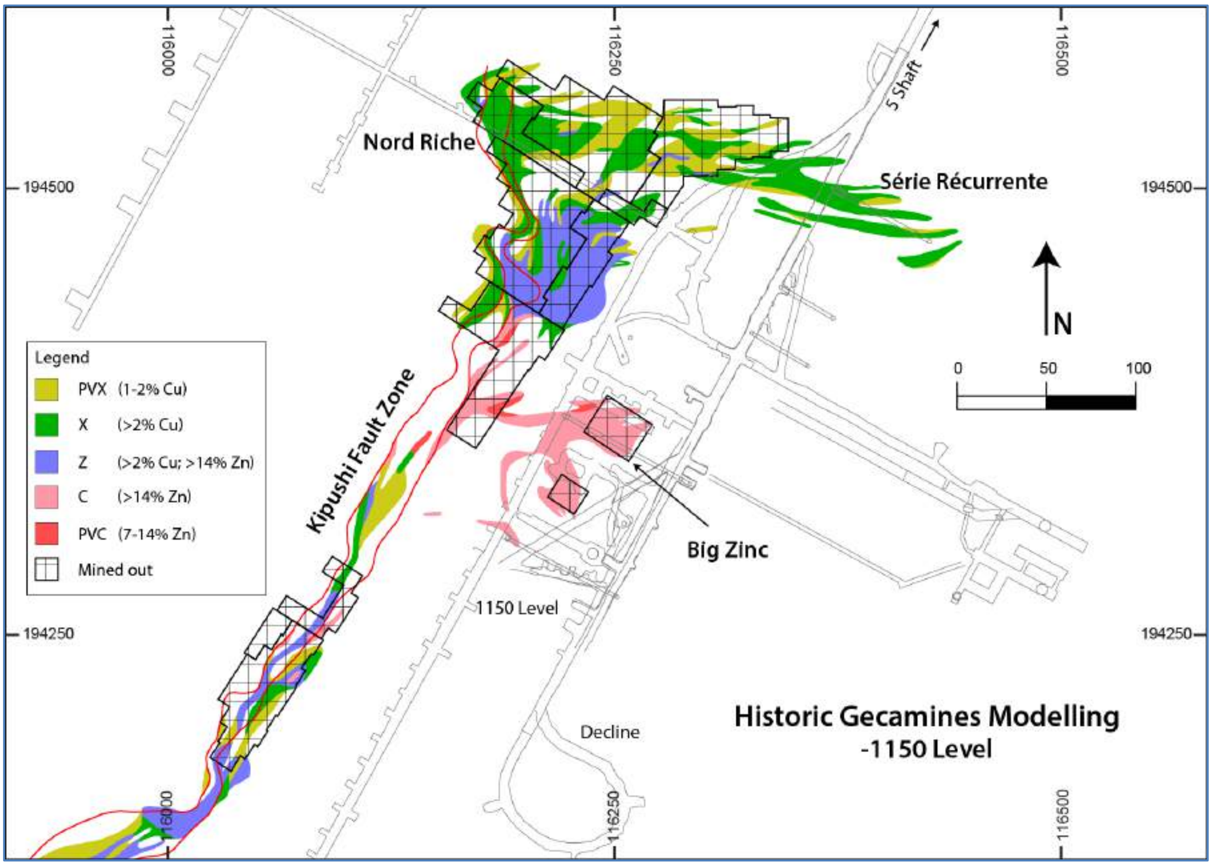


Figure by Ivanhoe, 2015.

6.4 Historical Production

The Kipushi deposit has largely been mined from surface down to approximately the 1,150 mRL. 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 tonnes of lead and approximately 278 tonnes of germanium in concentrate were produced. Historically, a zinc and copper concentrate was produced from sulphide 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.

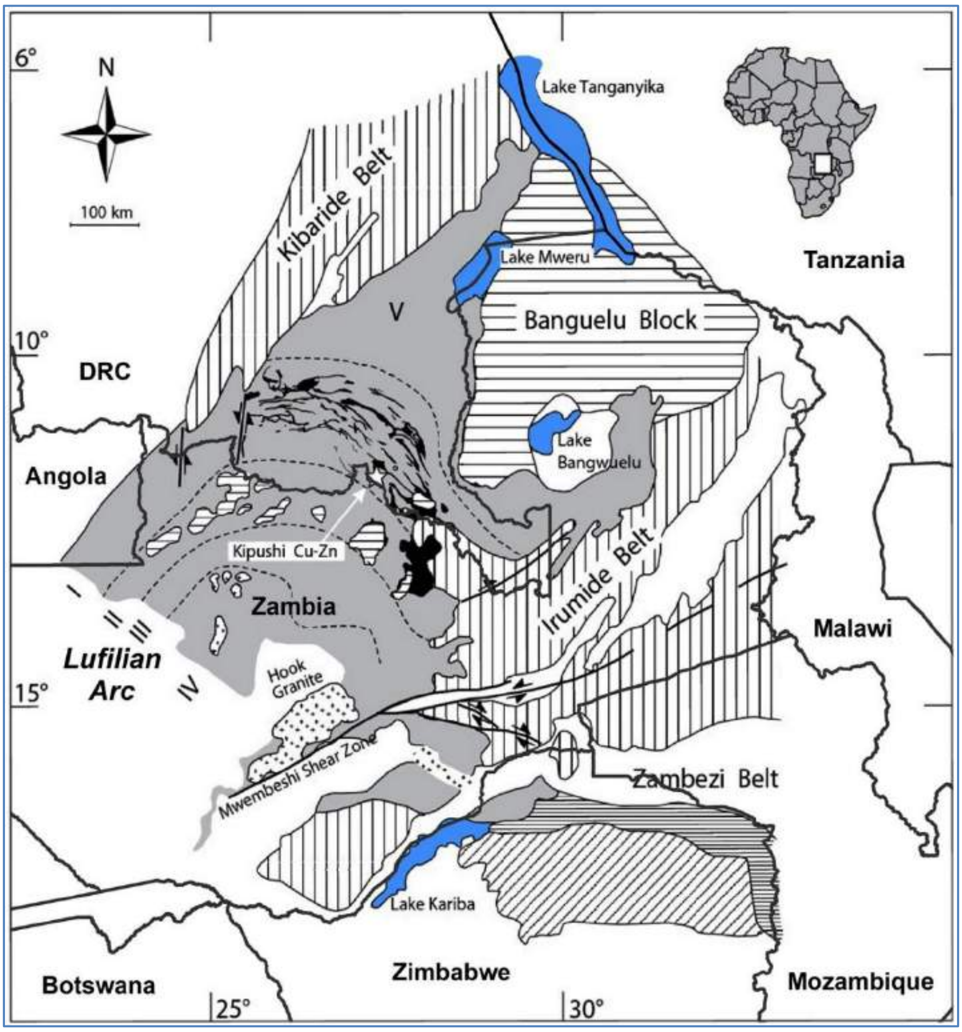
7 GEOLOGICAL SETTING AND MINERALISATION

The following review of the geological setting of the Kipushi Zn-Cu Project has been compiled from published literature as cited and as referenced in Section 27 of this Report, together with geological knowledge gained by KICO during the course of its underground drilling programme.

7.1 Regional Geology

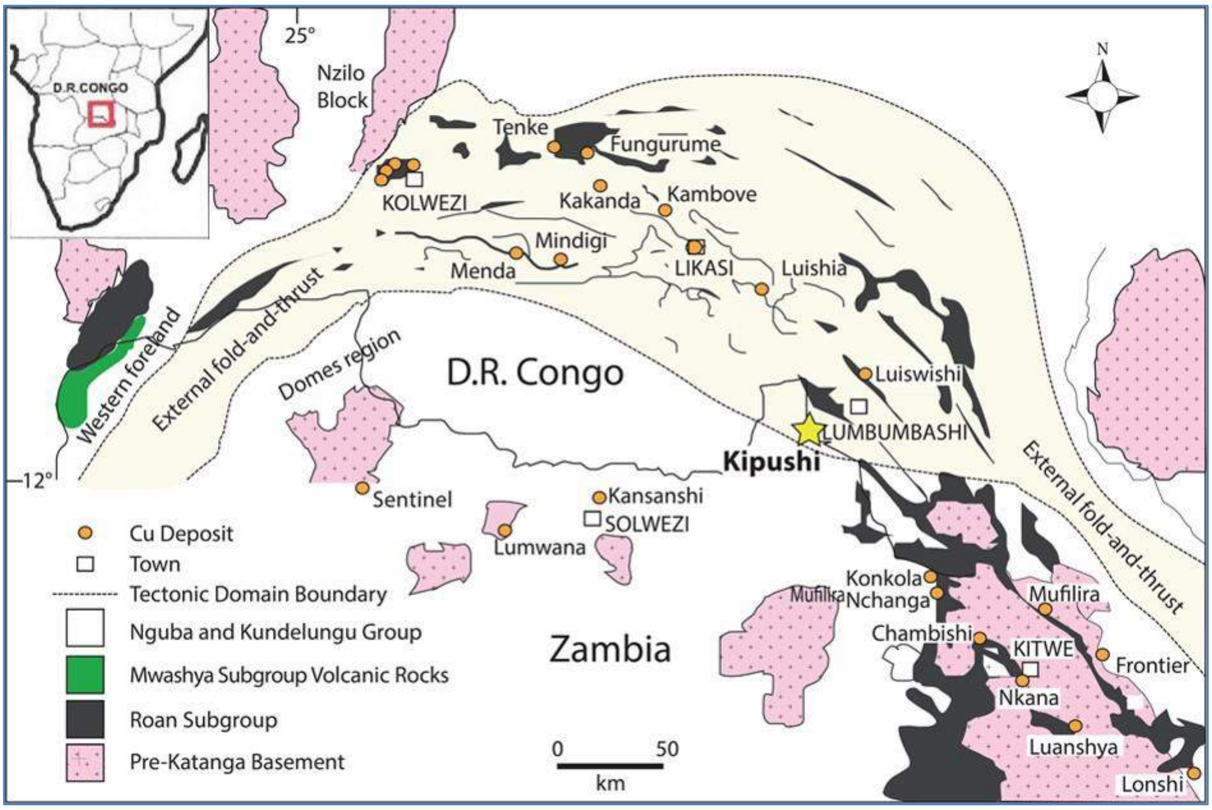
Kipushi is located within the Central African Copperbelt a northerly convex arc extending approximately 500 km from north central Zambia through the southern part of the DRC into Angola (Figure 7.1). The Central African Copperbelt constitutes a metallogenic province that hosts numerous world-class copper-cobalt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



Source: 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 Zn-Cu Project



Source: Ivanhoe Mines (2015) adapted after François (1974)

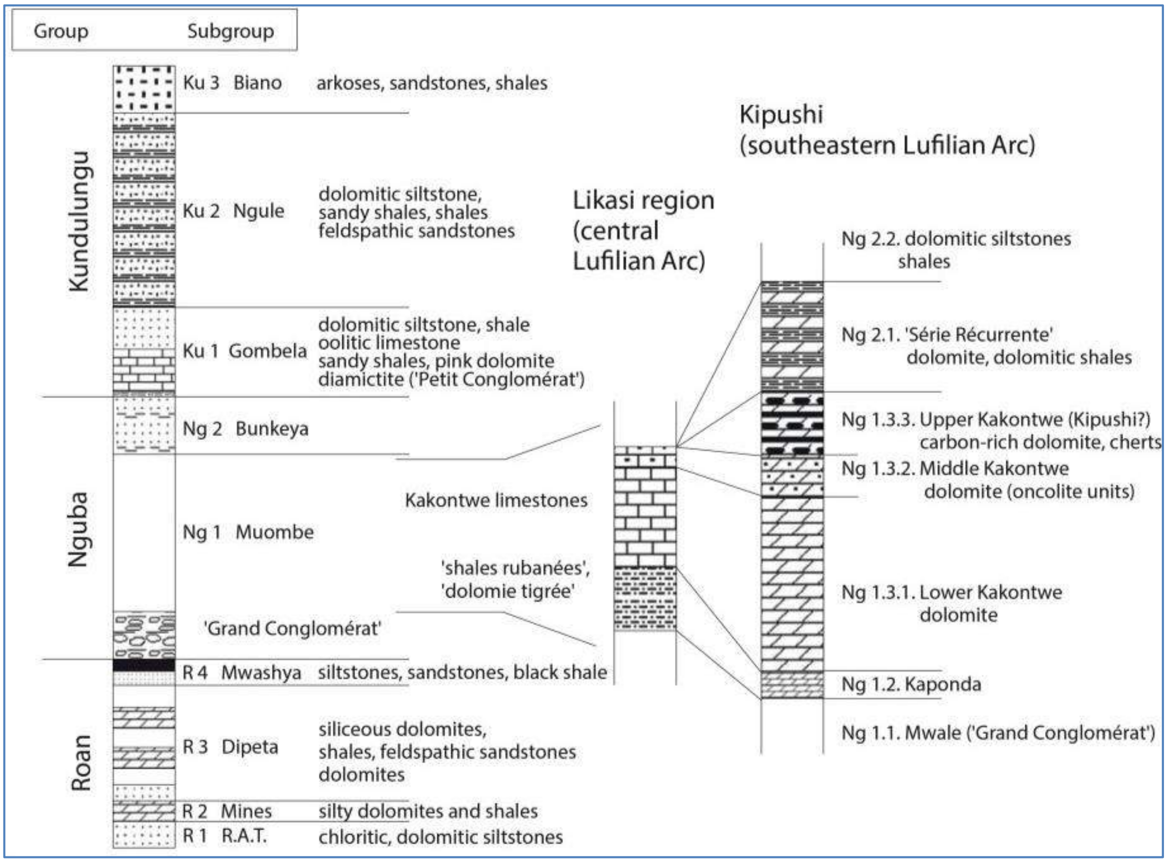
The Central African Copperbelt lies within the Lufilian Arc, a Pan-African age fold and thrust belt developed between the Congo Craton to the north–west and the Kalahari Craton to the south–east. The Lufilian Arc is one of several Neoproterozoic fold belts in Africa that originated through intracratonic rifting, sedimentation and subsequent closure accompanied by deformation and metamorphism. The Lufilian Orogeny involved north to north–eastward directed thrusting, leading to the formation of the northward convex Lufilian Arc. The crustal scale Mwembeshi Dislocation Zone separates the Lufilian Arc from the Zambezi Belt to the south.

The Lufilian Arc is composed of a 5–10 km thick sequence of metasedimentary rocks comprising the Katanga Supergroup. This is underlain by a basement comprising Neoarchaean granites and granulites 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



Source: Heijlen et al., (2008)

The Roan Group was deposited unconformably on the basement. The youngest included zircons in the basal sequence in Zambia give a maximum 880 Ma age for sedimentation (Armstrong, 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.] 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) mineralization 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 Tectonic Evolution

Sedimentation of the Katangan Supergroup began in a system of linked intracratonic rifts developed by the divergence and eventual break-up of the Rodinia Supercontinent (Selley et al., 2005). The transition from this initial syn-rift phase of continental deposition to a proto-oceanic rift basin is marked by the significant transgression of marine siliciclastics of the Mwashya Subgroup and overlying units of the Nguba and Kundulungu Groups over a wide area of the basin (Barron et al., 2003). The transition is also marked by the intrusion of tholeiitic mafic dykes in the Dipeta/Mwashya Subgroups, especially in northern Zambian (Barron et al., 2003) and extrusion of mafic and felsic tuffs (Kampunzu et al., 2000, Cailteux 1994).

A change from extensional tectonics to convergence occurs between 700 and 600 Ma (Cosi et al., 1992), however more recent dating constrains the Lufilian orogeny to between 600 Ma and 500 Ma, with the earliest dates (592 Ma) from greenschist-facies rocks in the Zambian Copperbelt (Rainaud et al., 2005). Deformation shows different expressions within concentric, northerly convex zones that parallel the Lufilian arc, with metamorphic grades increasing from the undeformed northern margins in the foreland, to the south.

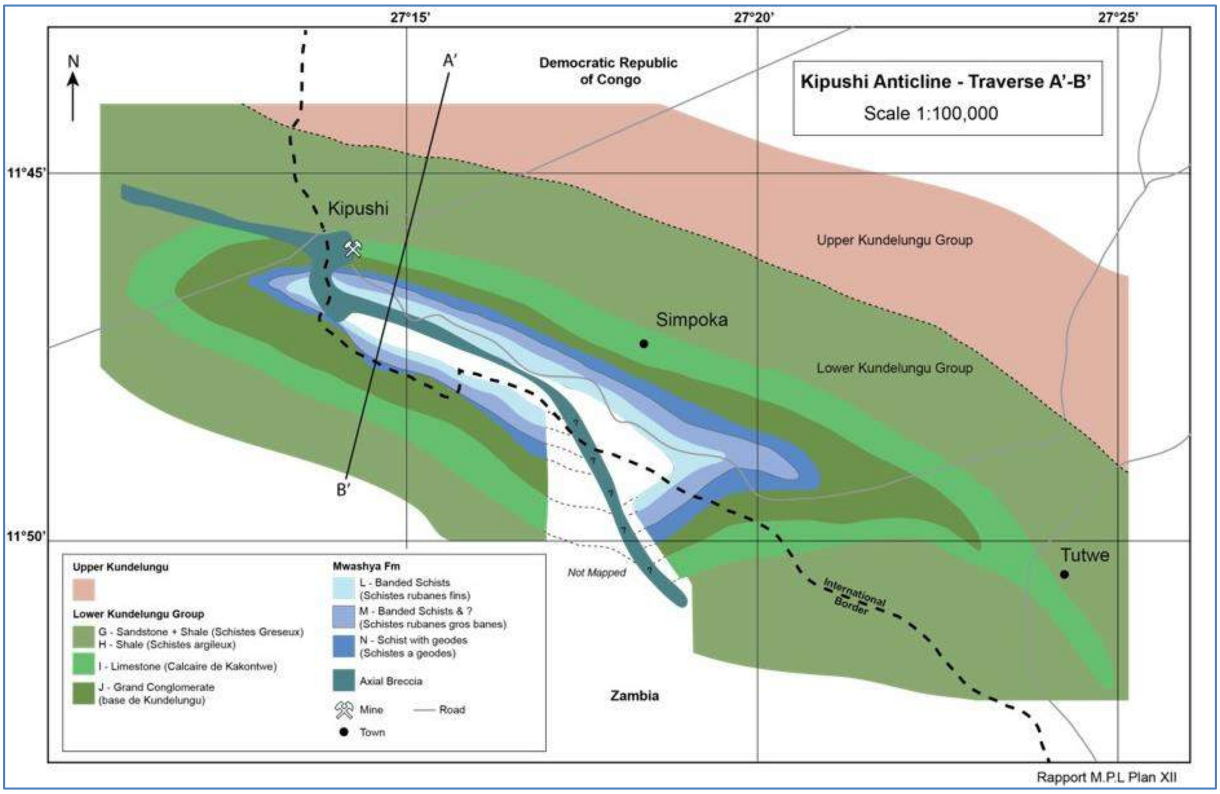
Unrug (1988) defined five structural domains within the Lufilian Arc: the external fold-and-thrust belt (I), the “Domes area” (II), the “Synclinorial belt” (III), the “Katangan High” (IV), and the “Katangan Aulacogen” (V). Kipushi occurs within the external fold and thrust belt as does the remainder of the Congolese Copperbelt, whereas the Zambian deposits occur adjacent to the easternmost basement inlier of the Domes region.

7.2 Local Geology

7.2.1 Structure

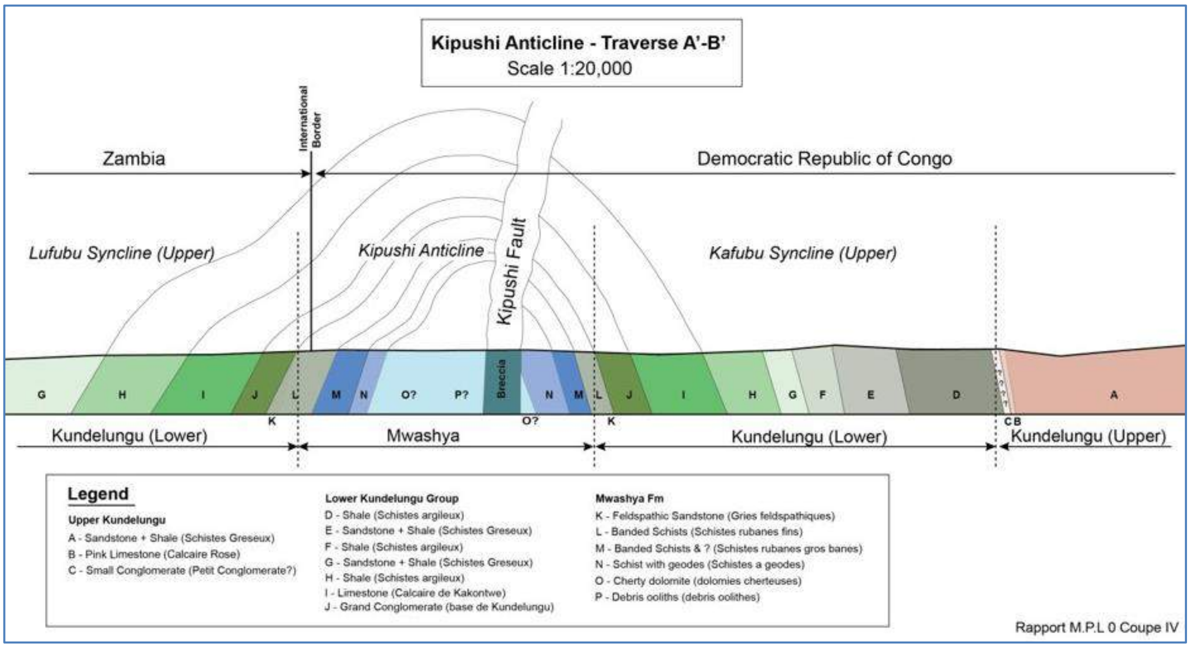
The Kipushi Zn-Cu Project is located on the northern limb of the regional west–north–west trending Kipushi Anticline which straddles the border between Zambia and the DRC. The northern limb of the anticline dips at 75–85° to the north–north–east and the southern limb at 60–70° to the south–south–west 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 that which typically underlies the thrusts related to the Lufilian Orogeny.

Figure 7.4 Geological Map of the Kipushi Anticline



Source: Ivanhoe Mines (2015) adapted after Briart (1947)

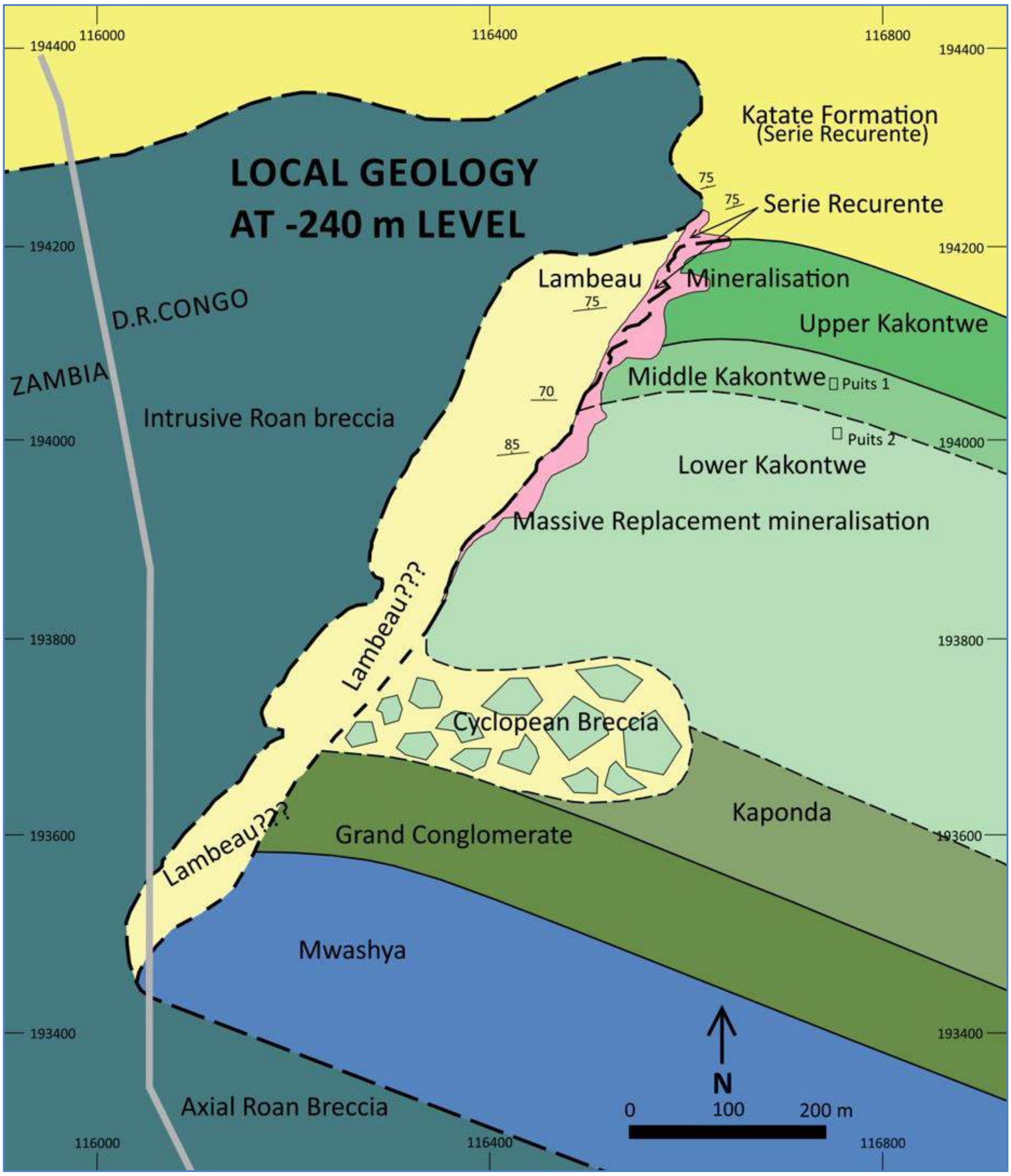
Figure 7.5 Section through the Kipushi Anticline



Source: Ivanhoe Mines (2015) adapted after Briart (1947)

An approximately north–north–east striking, approximately 70° west dipping discontinuity, several tens of metres in width and known as the ‘Kipushi Fault’ or ‘Kipushi Fault Zone’, juxtaposes Kakontwe strata to the east against a lens or block of generally barren siltstones and sandstones to the west. This lens is known locally as the “Grand Lambeau” (lambeau = fragment) and terminates the Kakontwe of the northern limb of the anticline against the fault zone on its footwall side Figure 7.6. The siltstones and sandstones of the Grand Lambeau are truncated on their western side by the intrusive axial breccia. The Kipushi Fault Zone has an irregular, highly sinuous geometry such that the location and orientation of its hangingwall and footwall contacts vary, commonly independently, along strike and down dip.

Figure 7.6 **Schematic Geological Map of the Kipushi Deposit at a Depth of 240 m below Surface. The Kakontwe Formation is Truncated Against a Syn-sedimentary Fault**

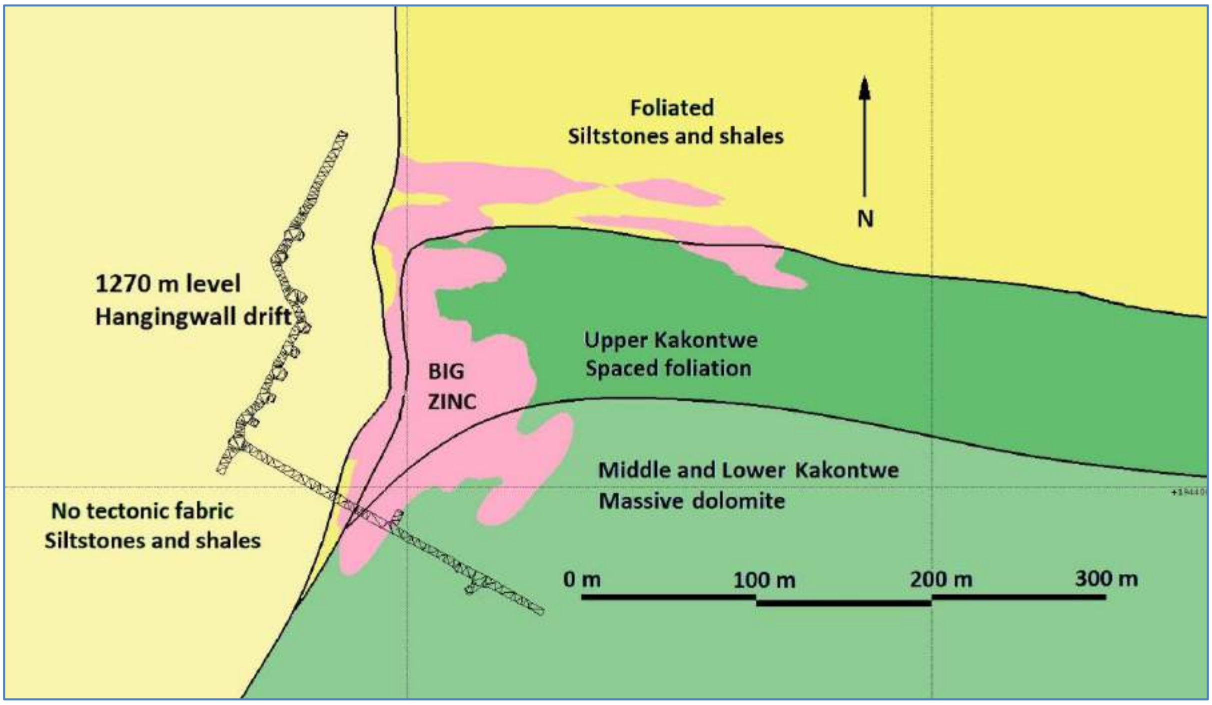


Source: Ivanhoe Mines (2015) adapted after Briart (1947)

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. Remarkably this configuration changes little in section, down to at least 1,200 m depth.

The Kipushi deposit is focused at the intersection of the Kakontwe and Katete Formations with the Kipushi Fault. Both formations maintain a uniform west–north–west/east–south–east strike along the northern flank of the Kipushi anticline, however, within 100 m of the fault zone the strike of the Upper Kakontwe and Katete formations begins to rotate towards parallelism with the fault zone. The juxtaposition of massive dolomites on the footwall side of a north–north–westerly trending syn-sedimentary fault, against siltstones on the hangingwall side, in-turn succeeded by siltstones and siltstone stratigraphically succeeding the dolomites gives a permanent rheological discontinuity that was multiply reactivated as the Kipushi Fault.

Figure 7.7 Kipushi Mineralization is Spatially Associated with the Rheological Contact between a Dolomite-Dominated Package to the South–east and Siltstones and Shales to the West and North



Source: Ivanhoe Mines (2015)

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 (Figure 7.8), both believed to have developed during north–north–east directed compression. Cleavage is close to parallel with bedding, over 100 m west of the fault zone. Towards the fault zone however, cleavage cuts bedding at an increasing angle.

Figure 7.8 Incipient Development of an Anastomosing Spaced Cleavage in the Upper Kakotwe Formation Looking West on a Footwall Drive on -865 m Level. Foliation can be seen to step down-stratigraphy (hence fabric steps down to the left in this photo)

Source: Ivanhoe Mines (2015)

Figure 7.9 Interbedded Dolomite-shale/Siltstone Unit in the Upper Kakontwe Formation at 153 m in KPU070 (hole orientation -35 to 125). Bedding Dips Steeply to NNW (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

Source: Ivanhoe Mines (2015)

7.2.2 Recent Work

Beyond the abundant literature focussing on mineralogy and geochemistry at Kipushi (e.g. Heijlen et al., 2008; Kampunzu et al., 2009, and references therein) there is 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 allocthonous 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.

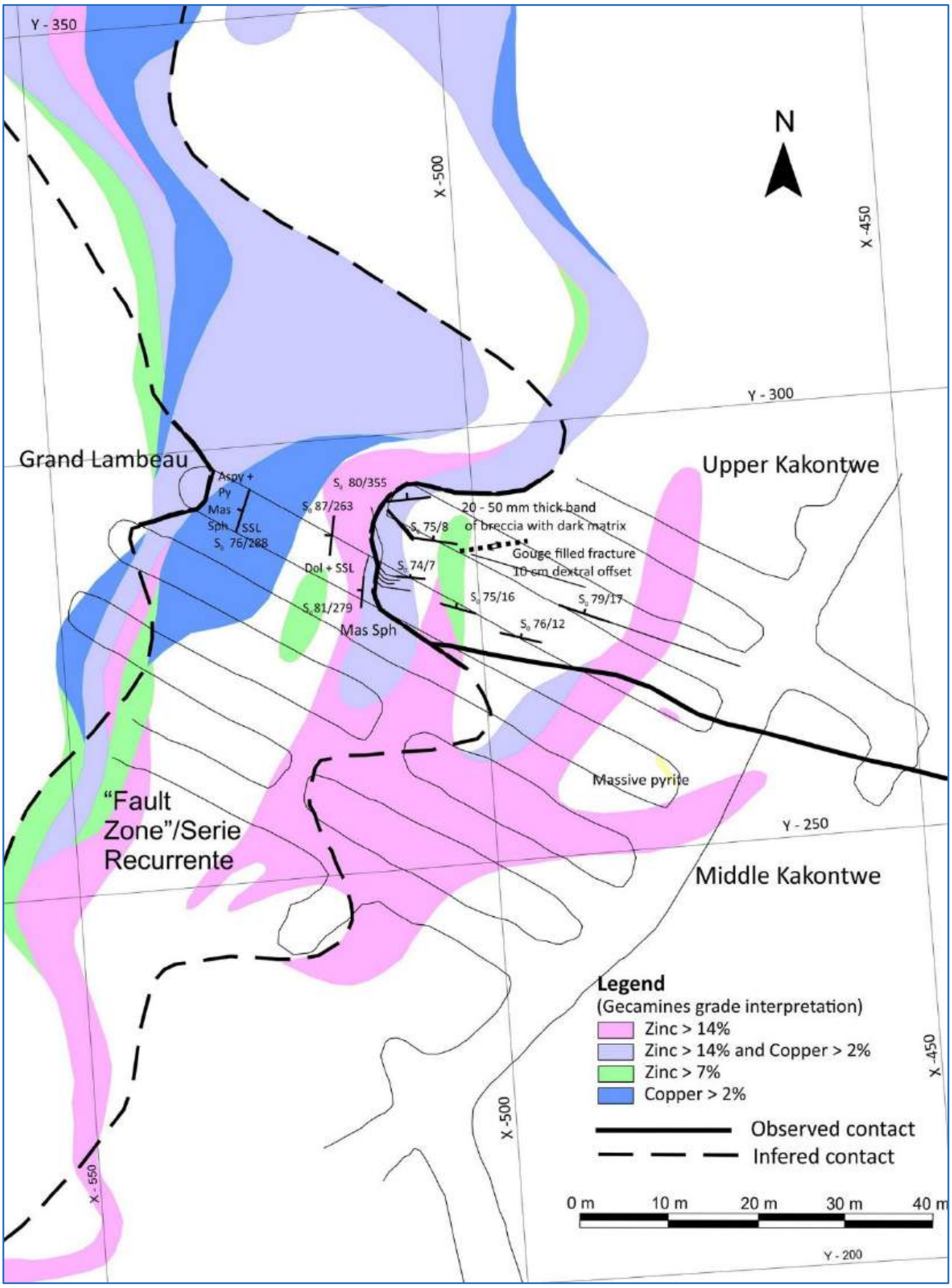
The only surviving production-era geological maps at Kipushi mine are level plans, on which structural data are few, mainly recording strike and dip and the upper contact of the Kakontwe Formation. Systematic underground mapping, if conducted, is no longer preserved, and surviving level plans and drill 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 re-interpretation of existing historical data, including drill cores.

Work by KICO currently envisages 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 fault zone has long been recognized as a locus for mineralization and this interpretation remains valid.

Observations from drilling and mapping on the 1,220 mRL and 1,275 mRL suggest a partly conformable stratigraphic succession exists across the northern side of the fault, between the Kakontwe and Katete Formations and the Grand Lambeau (Figure 7.10). This is especially clear in drillhole KPU074 (Figure 7.11), with siltstone and sandstone of the Grand Lambeau in partly conformable contact with siltstone and dolomite of the Série Récurrente at the level of the Upper Kakontwe and Série Récurrente. The local rotation of beds into parallelism with the fault zone has led to KICO re-interpreting this feature as a growth fault. Historical maps and sections also interpret a change in bedding orientation in close proximity to the fault. Although sections through the northern portion of the Kipushi Fault at the level of the Upper Kakontwe show an intact if condensed stratigraphy, more southerly sections at a lower stratigraphic level feature a modified stratigraphy with units that have been modified by carbonate dissolution during subsequent reactivation of the fault zone.

At the level of the upper Kakontwe Formation, the area of fault zone parallel to bedding coincides with a siltstone matrix supported sedimentary breccia with variously altered dolomite or shale clasts interpreted to be Série Récurrente that has slumped down the developing syn-sedimentary fault (Figure 7.12).

Figure 7.10 Mapping Undertaken by KICO on 1,220 mRL, with Grade Interpretations Taken from Historical Level Plans



Source: Ivanhoe Mines (2015)

Figure 7.11 Transition from Greenish-grey Siltstones of the Grand Lambeau (to 56.9 m) to the Purplish-grey Série Récurrente in Drillhole KPU074. This shows the Subtle Expression of the Northern Limit of the Kipushi Fault Zone

Source: Ivanhoe Mines (2015)

Figure 7.12 Sedimentary Breccia in the Kipushi Fault Zone from (top to bottom) KPU058 (77.4 m), KPU062 (82 m), KPU065 (87.8 m) and KPU066 (97.3 m). Pieces are 22 cm Long and Colour has been Enhanced

Source: Ivanhoe Mines (2015)

7.2.3 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.

A description of the Kipushi stratigraphy and traditional alpha-numeric nomenclature is given in Table 7.1. This coding method was maintained by KICO during the logging campaign.

Table 7.1 Revised Stratigraphic Column for the Kipushi Zn-Cu Project

Subgroup		Formation		Lithology (Hanging wall side)	Lithology (Footwall side)	Traditional Congolese designation	Mineralization
Upper Nguba (Bunkeya)	Monwezi	Katete Formation *(Série Récurrente)*		Fine grained sandstones, siltstones and minor calc-arenites of the '*Grand Lambeau*' Timing Uncertain.	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), characterized by black 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	Discordant 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 clotted calcimicrobial carbonates with a variety of textures. ~250 m thick.	Ki1.2.2.1	Discordant massive and replacement sph with minor cpy
	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 lighter dolomicritic layers forming 'Dolomite de Tigre' (Tiger Dolomite) pattern.	Ki1.2.1	

Source: Ivanhoe Mines (2015)

The Kipushi Fault is a 10–50 m wide complex structure recording multiple styles of deformation and brecciation. In most places it comprises two distinct hangingwall and footwall structures (contacts) with an intervening central zone of siltstones, shales and minor dolomites, all of which separates the footwall Kakontwe Formation from the hangingwall Grand Lambeau. The architecture of a growth fault on its northern side, clearly seen in plan view, has been significantly modified by subsequent deformation and alteration. Northern sections through the fault show a clear intact succession from Upper Kakontwe, to Série Récurrente to Grand Lambeau. However the section is considerably more complex and narrower in the south, such that it has necessitated the development of a local stratigraphy (Table 7.2).

Table 7.2 Kipushi Fault Zone Stratigraphy, in Order from Hangingwall to Footwall

Stratigraphic unit	Lithology	Codes or traditional stratigraphic designation
Grand Lambeau (Hangingwall)	WNW striking, steeply NNE-dipping, north-younging sequence of interbedded siltstone, sandstone and minor conglomerate with abundant sandstone dykes and dewatering structures. Upper (northern) portion postulated to be stratigraphically equivalent to the Série Récurrente. Locally mineralized close to the Kipushi Fault.	GLB
Série Récurrente	Interbedded purple dolomite and green siltstone gradational to deformed breccia with dolomite clasts/fragments/boudins (often veined or silicified) bound in a green/grey siltstone matrix. Rarely seen in south. Locally mineralized with pyrite and chalcopyrite.	Ki2.1
Fault Zone Siltstone-shale	Westward-younging and coarsening sequence of interbedded carbonaceous shale and grey siltstone, grades up-section (westward) from thin-bedded shale-siltstone to massive thick-bedded siltstone. Commonly includes a grey dolomite bed near the top (adjacent to contact with Grand Lambeau). Rarely seen in the north. Abundant fine grained, locally massive pyrite, mineralized near Big Zinc contact with red sphalerite and pyrite.	FZSSL
Carbonaceous Matrix Breccia	Clast or matrix supported dissolution breccia with dark grey/black carbonaceous matrix. Clasts of dolomite or siltstone (dolomite clasts are frequently embayed) depending on protolith. Rarely seen in the north. Often mineralized on Big Zinc contact with pyrite and red sphalerite.	CBX
Kakontwe formation (Footwall)	Intact middle or upper Kakontwe. Often carbon-bearing immediately next to fault-zone (where not replaced by sulphides).	Ki1.2.2.2/Ki1.2.2.3

Source: Ivanhoe Mines (2015)

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). Proceeding southwards along the fault zone, the volume of entrained higher stratigraphy diminishes as does the thickness of the fault zone.

Figure 7.13 A carbonaceous/argillaceous-matrix breccia in the Upper Kakontwe <100 m east of the Kipushi Fault Zone. The clasts are dolomite and chert fragments with some brassy pyrite. The extent of structural fabric development varies considerably indicating deformation postdates breccia formation. 610 m in KPU002

Source: Ivanhoe Mines (2015)

The contact between the Grand Lambeau and the Kipushi Fault Zone is marked by the following changes:

- The disappearance of economic mineralization - the Grand Lambeau locally hosts minor mineralization within several metres of the contact.
- A change from siltstones and carbonate in the fault zone to siltstones/sandstones
- Siltstones in the fault zone lack syn-sedimentary deformation textures, compared to abundant dewatering structures and sandstone dykes in the Grand Lambeau.
- A change in bedding orientation from ~north–north–east within the fault to west–north–west within the Grand Lambeau.

7.3 Alteration and Metamorphism

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

7.4 Mineralization

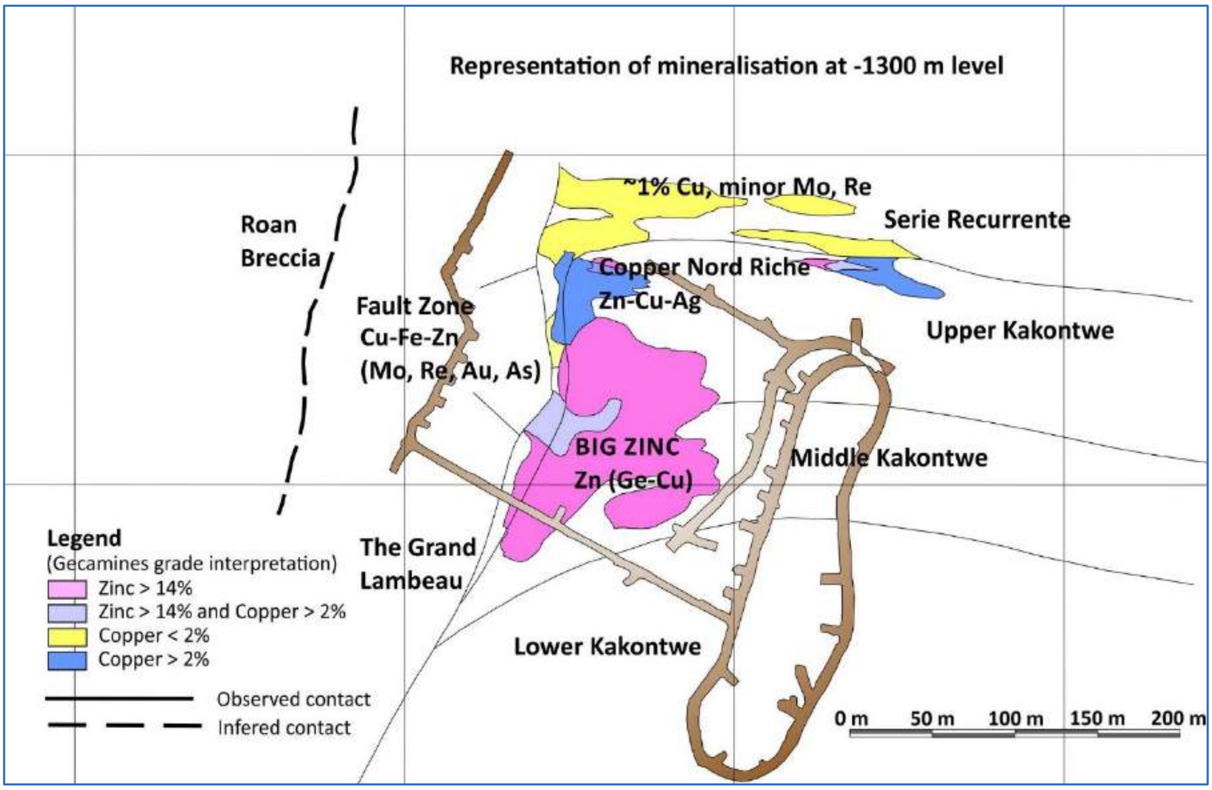
7.4.1 Overview

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 sulphide mineralization contained within dolomitic limestone (Kampunzu, et al., 2009). These deposits are polymetallic with a typical Zn-Pb-Cu-Ag-Cd-V association and also contain variable concentrations of As, Co, Mo, Rh, Ge, and Ga.

Mineralization at Kipushi is spatially associated with the intersection of Nguba Group stratigraphy with the Kipushi Fault and occurs in several distinct settings (Figure 7.14):

- Kipushi Fault Zone (copper, zinc and mixed copper-zinc mineralization both as massive sulphides and as veins),
- Série Récurrente zone (disseminated to veinlet-style copper sulphide mineralization),
- Upper Kakontwe zone (massive copper and zinc sulphides),
- Copper Nord Riche zone (mainly copper but also mixed copper-zinc sulphide mineralization, both massive and vein-style), and
- Big Zinc zone (massive zinc sulphide with local copper sulphide mineralization).

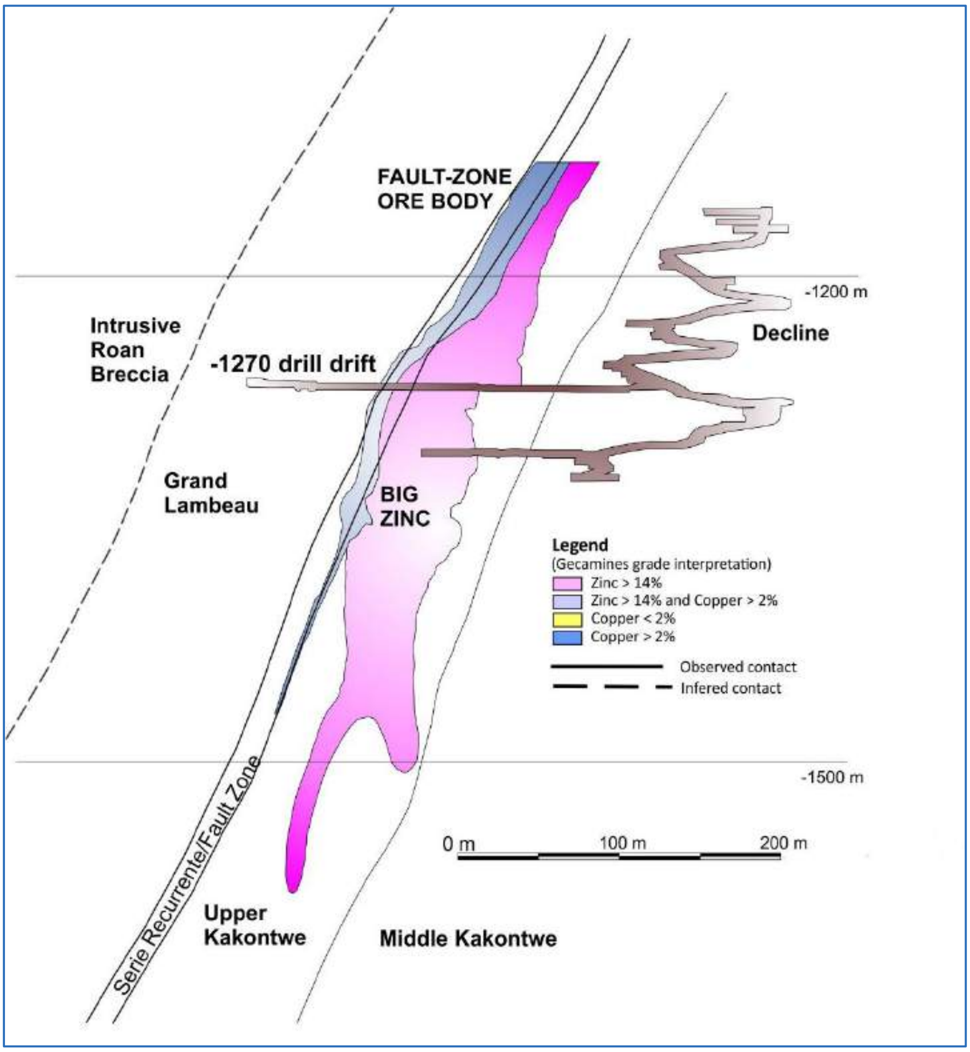
Figure 7.14 A representation of grade distribution at approximately 1,300 mRL. The grade classifications (and the colours) are consistent with those used on historical level plans and cross sections



Source: Ivanhoe Mines (2015)

Mineralization at the Kipushi Zn-Cu Project is generally copper-dominant or zinc-dominant with minor areas of mixed copper-zinc mineralization. Pyrite is present in some peripheral zones and forms massive lenses, particularly in the Kipushi Fault Zone. Copper-dominant mineralization in the form of chalcopyrite, bornite, and tennantite is characteristically associated with dolomitic shales both within the Kipushi Fault Zone and extending eastwards along, and parallel to, bedding planes within the Katete Formation (Série Récurrente). Zinc-dominant mineralization in the Kakontwe Formation occurs as massive, irregular, discordant pipe-like bodies completely replacing the dolomite host and exhibiting a structural control. 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 7.15).

Figure 7.15 Cross-section Perpendicular to the Kipushi Fault, Looking North–north–east



Source: Ivanhoe Mines (2015)

There is considerable variety in the mineralized zones and different styles sometimes occur with a diverse range of economically significant accessory minerals for which Kipushi is well known. Although the complex mineralogy of Kipushi has been documented for over 60 years, the lower levels of the deposit considered in this Kipushi 2016 PEA show simpler mineralogy.

Sulphide mineralization in the Kakontwe Formation occurs as massive, irregular, discordant pipe-like bodies completely replacing the dolomite host and exhibiting a structural control. The overlying Série Récurrente and Fault Zone host foliation-parallel sulphides as discontinuous lenticles or veinlets in foliated siltstone, and veins or local replacement in the interbedded massive dolomite. Mineralized zones in all Kakontwe units exhibit a steep southerly plunge from the Série Récurrente contact, or the Fault Zone, to their terminations in the footwall.

They also show a clear zonation from copper-rich at the Série Récurrente or Fault Zone contact, to zinc-rich to zinc- and pyrite-rich at their footwall terminations. The steep southerly plunge of the pods is difficult to reconcile with the intersection of the Upper Kakontwe and the Fault Zone giving a general north–west plunge to the Kipushi deposit.

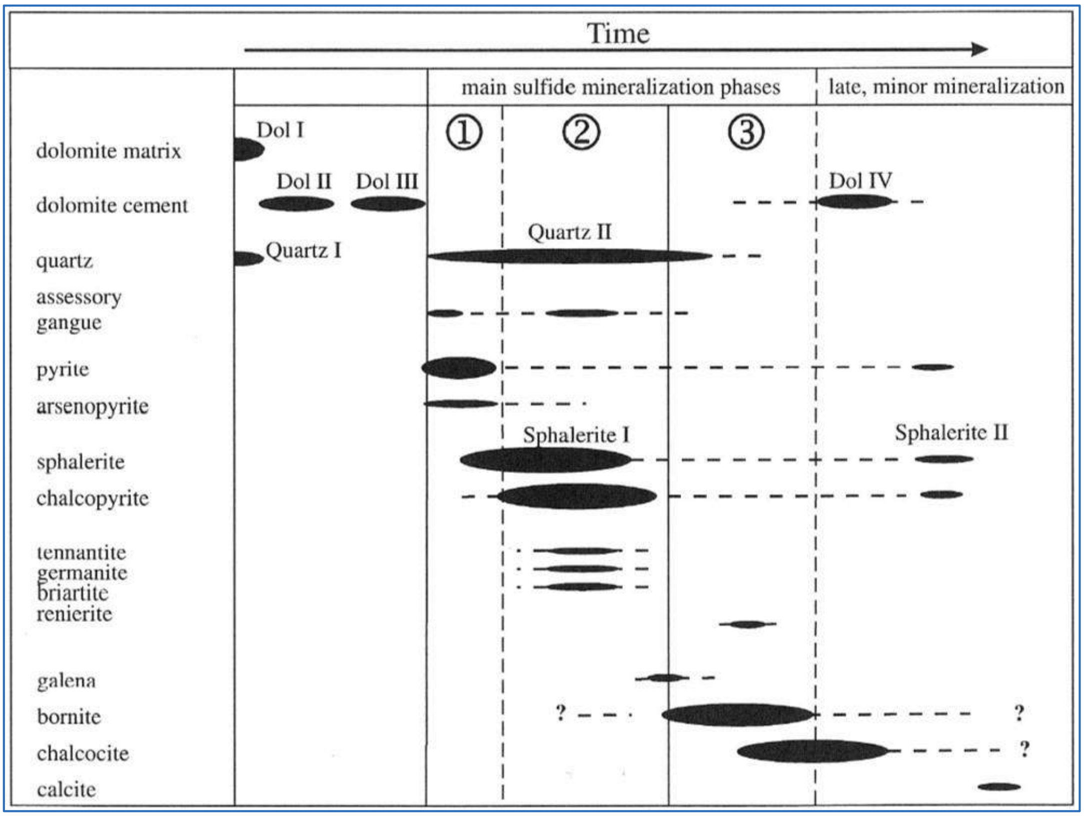
This mineral zonation is similar to that seen in other Central African Copperbelt deposits, wherein copper is proximal to source (for example, the Kipushi Fault Zone) whereas zinc and pyrite are distal.

Previous studies on the Kipushi mineralization have shown that the sulphide mineralization is complex and multiphase (e.g. Heijlen et al., 2008). 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 7.16. As a typical feature, mineralization 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 sulphide phase of pyrite-arsenopyrite mineralization was followed by sphalerite, chalcopyrite, tennantite, germanite, briartite and galena in a second major phase of sulphide deposition. As a third major phase, bornite and chalcocite appear to selectively replace chalcopyrite, as secondary mineralization in the higher levels of the mine.

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

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

Figure 7.16 Generalised Paragenesis of Mineralisation and Gangue at Kipushi



Source: Heijlen et al., (2008)

7.4.2 Kipushi Fault Zone

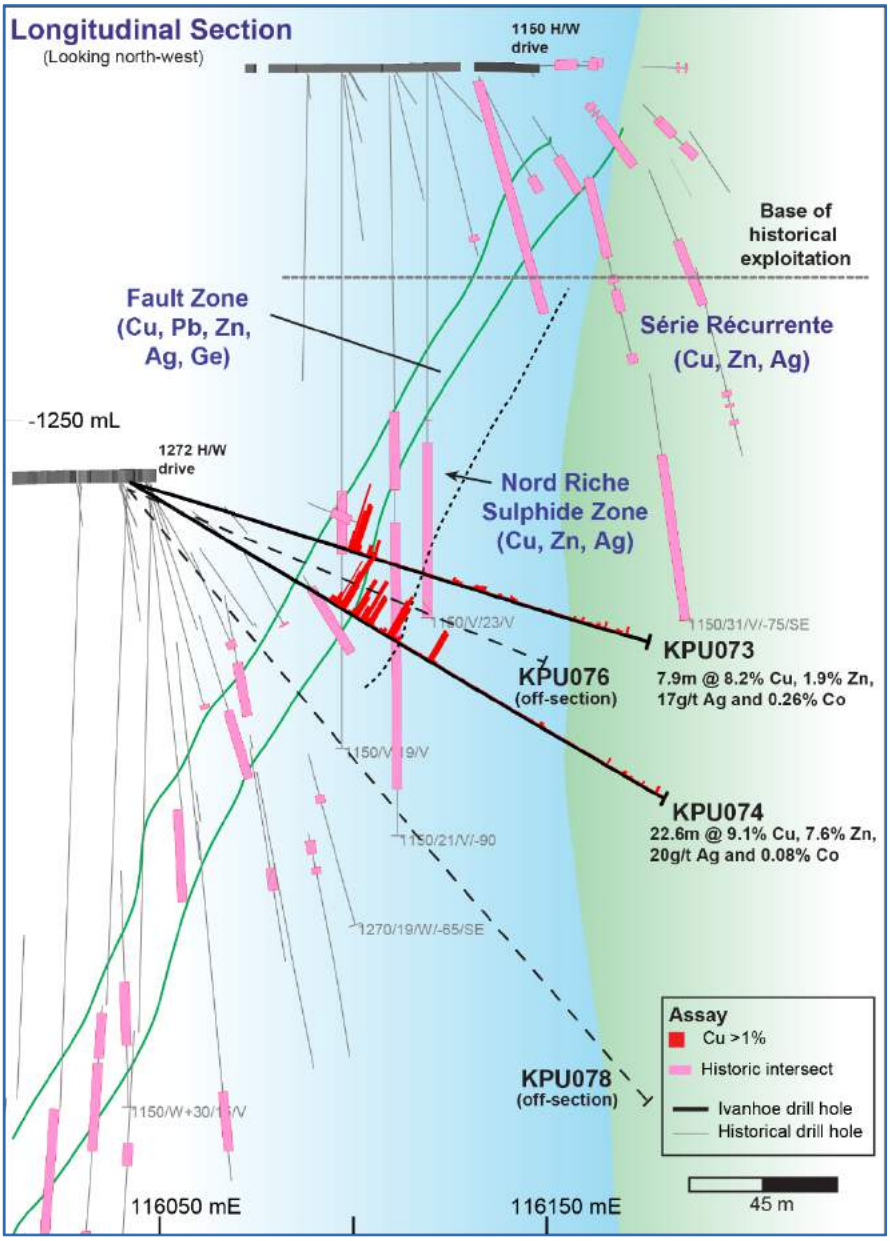
The Kipushi Fault Zone comprises Cu-Zn-Pb-Ag-Ge mineralization developed along the steeply north–west dipping Kipushi Fault between the Grand Lambeau to the west and intact Nguba Group stratigraphy to the east. Mineralisation locally extends laterally as discordant offshoots into rocks of the Kipushi (Upper Kakontwe) and Katete Formations in the footwall to the Kipushi Fault and terminates to the south–west where the Kipushi Fault 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 m to 10 m. Copper grades in the historically mined North zone decrease with depth from a maximum of 15% Cu to an average of 2% Cu at cessation of operations in 1993 (Kelly et al., 2012). In contrast, zinc grades increase with depth. Below 1,100 mRL, the Fault Zone deposit diverges into a central zinc-copper-lead-rich branch and an external zinc-rich branch (the Big Zinc zone) as shown in Figure 7.15.

The Fault Zone features a diverse range of textures, lithologies, and mineralization styles and types. The grade is variable and decreases southwards down-stratigraphy. Copper is prevalent in the Katete (Série Récurrente) Formation of the Fault Zone, which in southern sections exists near the hangingwall side of the Fault Zone. It resembles copper mineralization in the intact Katete formation, except that it is more pyritic and its associated albite-dolomite alteration is more intense. Between approximately 1,200–1,350 mRL, Big Zinc mineralization contacts the Fault Zone, where it is partially replaced with sphalerite and pyrite (Figure 7.18). It is postulated that sphalerite replaced the carbonate fraction of the fault-zone sedimentary/tectonic breccias. Immediately south of the Big Zinc zone, semi-massive chalcopyrite mineralization exists in the Middle and Upper Kakontwe dolomites in the immediate footwall to the Fault Zone, where higher stratigraphy has slumped toward parallelism with the contact.

Alteration associated with mineralization includes dolomitisation of the Kakontwe Formation limestone up to 200 m away from the deposit, silicification of wallrock dolomite, formation of black amorphous organic matter in the footwall dolomite up to 40 m away, chloritisation along mineralization contacts and along fractures, and kaolinisation of feldspars within the Grand Lambeau.

Figure 7.17 Longitudinal section at the northern end of the Kipushi Fault looking north–west and showing Fault Zone, Nord Riche and Série Récurrente mineralization together with historical and some recent drilling



Source: Ivanhoe Mines (2015)

Figure 7.18 An intercept of the Kipushi Fault Zone in KPU053. The contact with the hangingwall side of the Big Zinc is at 105.8 m

Source: Ivanhoe Mines (2015)

7.4.3 Copper Nord Riche Zone

Discreet mineralized zones of patchy to massive chalcopyrite mineralization with minor sphalerite are focussed at the top of the Upper Kakontwe Formation near its contact with the Katete Formation (Série Récurrente) in a zone known locally as the Copper Nord Riche (Figure 7.17). Mineralization in the Copper Nord Riche zone is significantly thicker than in the adjacent Série Récurrente zone. In the Copper Nord Riche zone, mineralized 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 7.19). Chalcopyrite intercepts frequently contain abundant silver (>100 ppm), arsenic (>5000 ppm) and molybdenum (>100 ppm), associated with tennantite.

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

The Copper Nord Riche area has been incompletely explored below the previous workings.

Figure 7.19 Drillhole KPU032 showing massive and patchy chalcopyrite/sphalerite mineralization in the Upper Kakontwe near the northern limit of the Fault Zone

Source: Ivanhoe Mines (2015)

7.4.4 Série Récurrente Zone

Disseminated chalcopyrite-bornite mineralization within alternating siltstones and dolomite beds of the Série Récurrente zone (Figure 7.20) extends from the Fault Zone to at least 150 m eastward along strike. Grades are generally around 1–2% Cu. This grade of mineralization extends from the Upper Kakontwe Formation contact 20 m into the Série Récurrente zone and gradually diminishes with increasing distance from the contact (Figure 7.17). Bornite tends to become more abundant than chalcopyrite northwards from the contact, suggesting an increase in Cu:S ratio, however, bornite tends to be localised in dolomite beds whereas chalcopyrite dominates in siltstone beds where it occurs with trace Mo and Re. Mineralization is best developed in siltstone, where it occurs as discrete 2–5 mm thick discontinuous veinlets or lenticles parallel or subparallel to foliation/bedding (Figure 7.21). 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. Mineralization in dolomite is also vein-hosted, but without the strong structural control seen in the deformed siltstone. Chalcopyrite is best developed in reduced, siltstone beds were it occurs with trace Mo and Re.

Figure 7.20 Typical colour variation in the Série Récurrente between dolomite (purple) and siltstone (green)

Source: Ivanhoe Mines (2015)

Figure 7.21 Blebby and disseminated chalcopyrite in Série Récurrente siltstone at 148 m in drillhole KPU074. Both mineralization and bedding are deformed by parasitic folds

Source: Ivanhoe Mines (2015)

7.4.5 Big Zinc Zone

The Big Zinc is a zone of massive sphalerite mineralization in the Middle and Upper Kakontwe Formations in the immediate footwall to the Kipushi Fault Zone between 1,100–1,650 mRL. Mineralization is discordant and occurs at least 100 m laterally along the footwall of the fault and extends up to 80 m into the footwall near the contact between the Middle and Upper Kakontwe Formations. 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 zone (Figure 7.15). The zone diverges from the Kipushi Fault Zone with increasing depth.

The contacts of mineralization with the host Kakontwe dolomite are zoned over several metres as shown in Figure 7.22. Sphalerite on the margins of the mineralized zone, particularly at the terminations of apophyses, is often red and iron-rich (Figure 7.22) 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 the eastern and down-plunge margins. The outer (distal to the fault) contacts are commonly marked by an abundance of distinctive megacrystic and "mosaic-textured" white hydrothermal dolomite inter-grown with the sulphides (Figure 7.24).

The Big Zinc zone is mineralogically simple with the majority of the deposit comprising massive, monotonous equigranular to occasionally banded honey-brown sphalerite and pyrite (Figure 7.22). Mineralization 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 northern side of the deposit, in the Upper Kakontwe Formation, 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 zone and at depth, particularly in very zinc-rich sphalerite. There is nothing to visually distinguish the very high grade (>55% Zn) and germanium rich (>100 ppm Ge) sphalerite from the majority of sphalerite within the Big Zinc zone.

Tennantite, bornite and chalcopyrite locally replace sphalerite in a 10 to 20 m thick pod of >100 m plunge extent within the Big Zinc zone. Smaller zones of tennantite mineralization have been seen elsewhere in the Big Zinc and Copper Nord Riche zones. These zones are associated with very high silver, cobalt, molybdenum grades and elevated gold (Figure 7.23).

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 7.22 Mineralization intersected in historical drillhole 1270/15/-40/SE: A) Chalcopyrite-dominated Fault Zone, B) reddish iron-rich sphalerite on the margins of the Big Zinc, and C) honey-coloured sphalerite in the central part of the Big Zinc

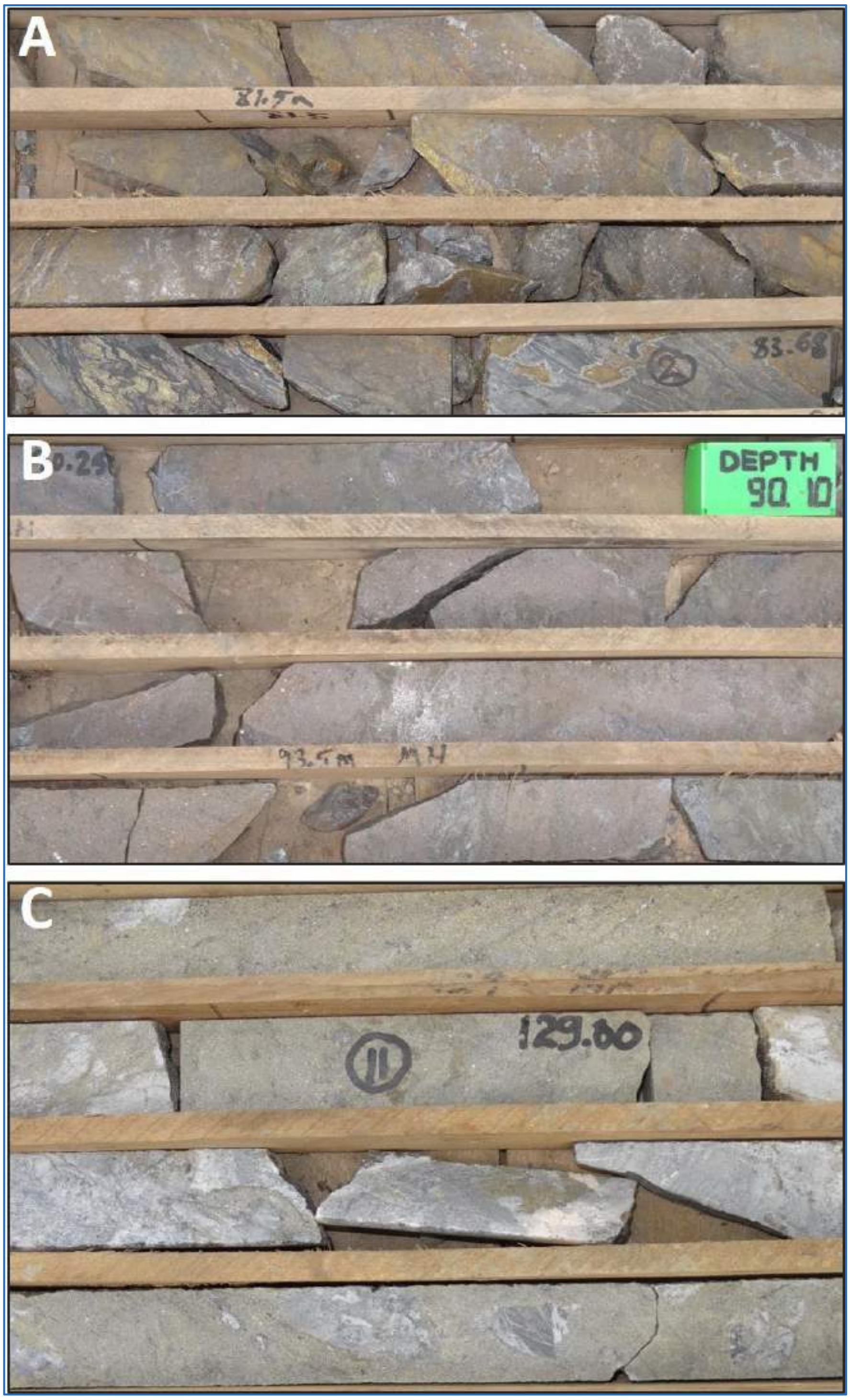
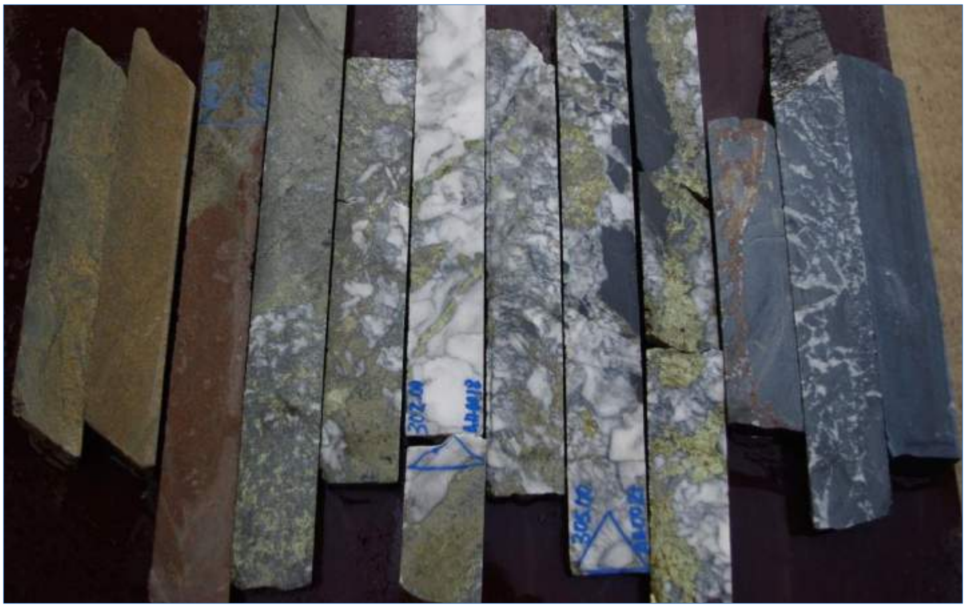


Figure 7.23 Mineral and metal zonation at the distal margin of the Big Zinc: ZnS > Zn(Fe)S ± PbS > Cpy > Pyrite ± Aspy > from the Big Zinc (left) to the footwall contact. Note the distinctive mosaic-textured megacrystic mineralisation -stage dolomite.



Source: Ivanhoe Mines (2015)

Figure 7.24 Chalcopyrite-tennantite-bornite Replacement within the Big Zinc, Drillhole KPU040

Source: Ivanhoe Mines (2015)

8 DEPOSIT TYPES

The mineral deposits at Kipushi are an example of carbonate-hosted copper-zinc-lead mineralization hosted in pipe-like fault breccia zones, as well as tabular zones. This deposit type tends to form irregular, discordant mineralized bodies within carbonate or calcareous sediments, forming massive pods, breccia/fault fillings and stockworks (Trueman, 1998). They often form pipe-like to tabular deposits strongly elongate in one direction. Zinc-lead rich mantos can project from the main zone of mineralization 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, sabkha, 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. Mineralization forms as fault or breccia filling, and massive replacement mineralization with either abundant diagenetic pyrite or other source of sulphur (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 sulphide deposition. Deposits are usually coincident with a zone of dolomitisation. Pre-mineralization 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 dolomitisation, 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 sulphide deposits are hosted in deformed platform carbonates of the Lufilian Arc. In the 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 associated with collapse breccias and faults as well as lenticular bodies subparallel to stratigraphy. They 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 sulphide deposition within favourable structures and reactive carbonate sequences. In the case of the Big Zinc zone, massive sphalerite mineralization 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 KICO on the Kipushi Zn-Cu Project.

10 DRILLING

10.1 Historical Drilling

10.1.1 Drilling Methodology

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 zone and 12.5 m apart along the Série Récurrente zone, 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 Section 0 to 19 along a 285 m strike length including a 100–130 m strike length which also tested the Big Zinc zone.

Drill core from 49 of the 60 holes drilled from 1,272 mRL that intersected the Big Zinc zone 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 mineralized 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 sphalerite mineralization in the form of veins and disseminations.

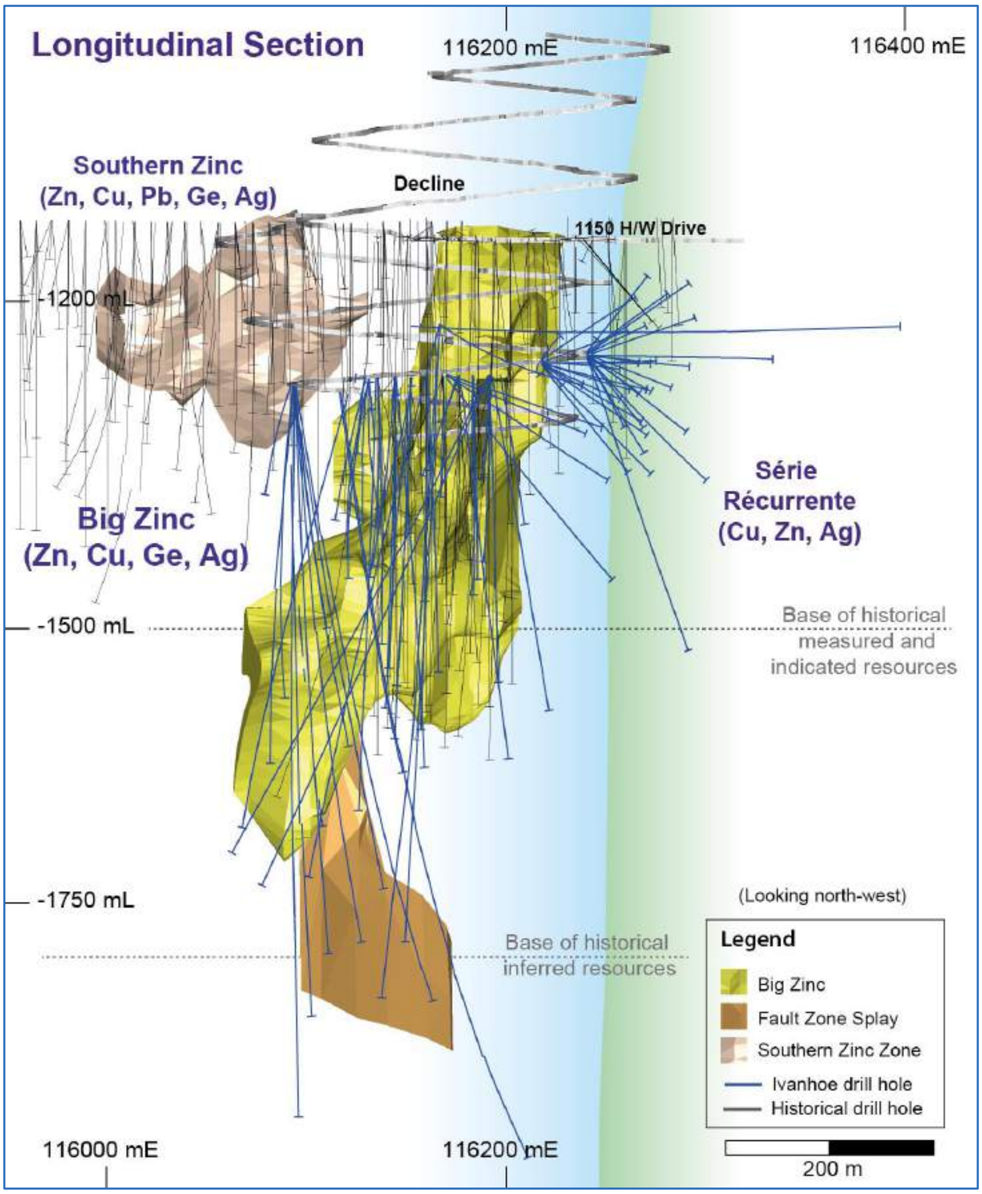
The drilling methodology is described in Kelly et al., (2012). On completion of each hole, collar and downhole surveys were conducted and the following information was recorded on drill log sheets:

- 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; and
- Hydrology and temperature.

Some of the drill log sheets contained missing information (Kelly et al., 2012).

A total of 84 holes intersected the Big Zinc zone of which 55 holes were surveyed downhole at a nominal 50 m spacing. Gécamines sampling tended to be based on lengths representing mineable zones, with little attention paid to geology and mineralization (Kelly et al., 2012).

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



Source: Ivanhoe Mines (2015)

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 mineralization log, as standardised simple codes;

- Downhole survey data; and
- 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, 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 Zn-Cu 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, a 25,400 m underground drilling programme was carried out by KICO between March 2014 and October 2015, with the cut-off date of 16 December 2015 for data included in the Kipushi 2016 PEA.

The 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 drilling programme were to:

- Conduct confirmatory drilling to validate the Historical Estimate within Kipushi's Big Zinc zone 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 zone and Fault Zone below 1,500 mRL.
- Test for deeper extensions to the Big Zinc zone by drilling from the 1,272 mRL hangingwall 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 mineralization of the mine's Copper Nord Riche zone.
- Gain an improved understanding of geology and controls on mineralization.

Underground drilling of the various mineralized zones was carried out from the footwall ramp and the hangingwall drive on 1,272 mRL. 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 operates 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 +/- 15°. Drilling has been completed from the 1,272 mRL drill drive along the Kipushi Fault Zone from Section 0 to 19 (see Figure 6.1 for section lines) and along a 285 m strike length, including a 100 to 130 m strike length in the vicinity of the footwall of the Big Zinc zone. Further north–east along the Kipushi Fault Zone, drilling from the same level has been partially completed along a 30 m strike length between Sections 21 to 23.

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 zones (Figure 10.1). Along the section lines, the drillholes intersected mineralization between 10 m and 50 m apart within the Big Zinc zone and adjacent Fault Zone, and up to 100 m apart in the deeper parts of the Fault Zone.

As at the effective date of this report, a total of 97 holes had been drilled for 25,419 m including 51 holes that intersected the Big Zinc zone (Table 10.1). Drillhole locations are shown in Figure 10.1 and summary parameters in Table 10.1.

Table 10.1 Underground Drilling Summary

Hole ID	Easting (m)	Northing (m)	RL (m)	Depth (m)	Azimuth	Dip	Start Date	End Date
KPU001	116173.47	194400.09	-1221.51	613.70	298.23	-66.35	01/03/2014	19/03/2014
KPU002	116173.47	194400.09	-1221.51	732.40	298.79	-60.57	19/03/2014	02/04/2014
KPU003	116173.25	194400.11	-1220.91	587.40	272.35	-62.69	02/04/2014	11/04/2014
KPU004	116308.92	194474.12	-1250.46	167.50	6.67	-45.56	02/04/2014	09/04/2014
KPU005	116308.90	194473.66	-1250.75	290.60	4.13	-68.14	09/04/2014	26/04/2014
KPU006	116308.08	194473.85	-1250.66	116.80	344.63	-50.01	19/06/2014	24/06/2014
KPU007	116175.19	194367.83	-1221.88	453.10	40.16	-89.98	15/04/2014	23/05/2014
KPU008	116308.97	194474.07	-1249.14	105.60	4.60	-5.21	26/04/2014	05/05/2014
KPU009	116309.02	194473.81	-1247.10	98.80	8.13	36.53	09/05/2014	03/06/2014
KPU010	116173.15	194400.73	-1220.30	245.80	330.17	-19.24	26/05/2014	31/05/2014
KPU011	116308.96	194474.06	-1249.54	74.20	3.96	-25.97	03/06/2014	05/06/2014
KPU012	116308.97	194473.90	-1250.75	74.80	7.49	-59.34	05/06/2014	07/06/2014
KPU013	116308.09	194473.85	-1249.97	101.80	347.04	-42.82	09/06/2014	11/06/2014
KPU014	116308.12	194473.78	-1249.54	80.80	343.81	-22.94	12/06/2014	13/06/2014
KPU015	116308.14	194473.76	-1249.05	71.80	342.82	-3.51	13/06/2014	15/06/2014
KPU016	116308.12	194473.75	-1248.07	83.80	346.78	-20.55	15/06/2014	17/06/2014
KPU017	116308.25	194473.28	-1246.90	19.50	343.03	40.26	18/06/2014	19/06/2014
KPU018	116308.32	194473.33	-1246.92	110.70	344.49	40.62	26/06/2014	29/06/2014
KPU019	116312.30	194475.07	-1246.63	110.80	35.92	38.23	03/07/2014	05/07/2014
KPU020	116312.60	194475.60	-1247.76	101.10	35.55	18.41	04/07/2014	08/07/2014
KPU021	116313.32	194476.85	-1250.43	77.80	32.94	-19.46	08/07/2014	10/07/2014
KPU022	116194.75	194309.20	-1271.31	41.00	307.44	-42.44	09/07/2014	11/08/2014
KPU023	116312.70	194475.80	-1250.60	110.80	35.92	-38.03	11/07/2014	14/07/2014
KPU024	116194.48	194309.26	-1271.20	5.80	307.53	-28.75	11/07/2014	11/07/2014
KPU025	116194.59	194309.16	-1271.46	83.80	310.26	-39.55	11/07/2014	15/07/2014
KPU026	116313.01	194476.21	-1249.02	166.70	35.40	-1.50	14/07/2014	19/07/2014
KPU027	116194.45	194309.35	-1271.09	251.80	303.55	-28.93	15/07/2014	23/07/2014
KPU028	116237.92	194467.66	-1255.88	230.60	296.27	-4.74	21/07/2014	08/08/2014
KPU029	116194.37	194309.02	-1270.74	251.80	295.90	-29.48	24/07/2014	28/07/2014
KPU030	116194.45	194309.20	-1270.79	302.80	292.97	-30.77	29/08/2014	31/08/2014
KPU031	116194.42	194309.06	-1270.93	299.80	294.36	-35.50	31/07/2014	08/08/2014
KPU032	116238.25	194467.63	-1255.31	221.80	304.61	4.13	08/08/2014	16/08/2014
KPU033	116136.06	194343.44	-1270.12	140.70	296.05	-31.33	14/08/2014	16/08/2014
KPU034	116136.20	194343.37	-1269.19	101.80	296.15	-0.91	16/08/2014	18/08/2014
KPU035	116239.19	194468.87	-1256.93	38.80	30.86	-0.31	16/08/2014	18/09/2014
KPU036	116239.15	194468.87	-1256.26	131.80	334.68	-15.85	18/08/2014	22/08/2014
KPU037	116135.88	194343.15	-1270.58	182.80	284.03	-40.32	19/08/2014	25/08/2014
KPU038	116239.00	194469.10	-1255.81	131.80	334.37	-2.09	22/08/2014	26/08/2014
KPU039	116241.03	194467.60	-1255.67	128.80	356.63	-0.37	27/08/2014	29/08/2014
KPU040	116013.70	194436.16	-1269.42	266.80	118.76	-65.85	27/08/2014	31/08/2014
KPU041	116241.04	194467.80	-1255.94	101.80	357.26	-13.54	30/08/2014	02/09/2014
KPU042	116013.98	194435.95	-1269.44	230.80	120.30	-52.50	01/09/2014	05/09/2014
KPU043	116240.92	194467.81	-1255.20	101.80	354.74	15.57	02/09/2014	05/09/2014
KPU044	116240.91	194467.10	-1254.49	122.90	353.84	28.97	05/09/2014	08/09/2014
KPU045	116242.74	194466.82	-1256.72	107.90	22.54	-28.57	09/09/2014	12/09/2014
KPU046	116029.75	194463.65	-1269.05	200.80	133.33	-44.50	05/09/2014	14/09/2014
KPU047	116242.89	194467.23	-1255.82	102.10	21.00	-0.47	12/09/2014	16/09/2014
KPU048	116029.70	194462.60	-1269.35	8.80	122.00	-65.00	–	–
KPU049	116242.80	194466.99	-1256.31	101.80	20.29	-14.92	16/09/2014	18/09/2014
KPU050	116028.21	194463.29	-1269.34	200.80	129.60	-50.10	16/09/2014	19/09/2014
KPU051	116027.70	194463.80	-1269.35	341.80	128.40	-75.50	21/09/2014	04/10/2014
KPU052	116243.35	194466.19	-1257.13	143.80	23.28	-44.49	21/09/2014	24/09/2014
KPU053	116242.32	194466.65	-1257.12	140.80	355.46	-46.67	25/09/2014	29/09/2014

Hole ID	Easting (m)	Northing (m)	RL (m)	Depth (m)	Azimuth	Dip	Start Date	End Date
KPU054	116242.26	194466.98	-1257.13	134.80	355.21	-29.11	30/10/2014	04/10/2014
KPU055	116028.13	194463.36	-1268.97	300.20	127.41	-69.21	04/10/2014	15/10/2014
KPU056	116035.23	194476.41	-1268.46	332.80	115.34	-76.85	05/10/2014	21/10/2014
KPU057	116022.04	194448.88	-1268.92	315.20	119.30	-73.90	17/10/2014	23/10/2014
KPU058	116042.88	194489.33	-1268.10	200.60	122.51	-36.22	23/10/2014	30/10/2014
KPU059	116022.39	194448.72	-1268.87	212.80	127.07	-54.17	23/10/2014	27/10/2014
KPU060	116022.98	194448.53	-1268.67	27.00	120.15	-38.73	27/10/2014	28/10/2014
KPU061	116023.26	194449.99	-1268.72	360.40	71.56	-81.90	29/10/2014	07/11/2014
KPU062	116022.73	194448.91	-1268.73	293.80	125.51	-65.56	30/10/2014	06/11/2014
KPU063	116040.98	194505.68	-1267.43	74.20	135.53	-34.49	07/11/2014	12/11/2014
KPU064	116023.41	194449.92	-1268.90	300.10	125.51	-65.56	09/11/2014	20/11/2014
KPU065	116042.10	194507.00	-1267.04	179.90	135.20	-27.17	12/11/2014	20/11/2014
KPU066	116029.30	194530.27	-1267.91	230.80	121.92	-61.06	21/11/2014	28/11/2014
KPU067	116012.38	194435.23	-1269.28	399.00	149.53	-83.15	21/11/2014	07/12/2014
KPU068	116029.99	194529.94	-1266.93	170.80	120.80	-19.77	28/11/2014	03/12/2014
KPU069	116037.31	194543.36	-1266.95	302.80	118.35	-69.11	04/12/2014	10/12/2014
KPU070	116038.61	194542.73	-1266.55	167.90	124.71	-34.32	10/12/2014	12/12/2014
KPU071	116035.20	194476.35	-1268.40	302.80	116.97	-61.23	08/01/2015	15/01/2015
KPU072	116037.23	194542.63	-1267.15	521.80	189.29	-64.41	09/01/2015	28/01/2015
KPU073	116043.98	194491.78	-1266.65	165.00	55.13	-17.58	16/01/2015	23/01/2015
KPU074	116044.06	194491.87	-1267.30	188.90	55.19	-30.47	24/01/2015	31/03/2015
KPU075	116037.43	194541.45	-1267.21	527.80	171.86	-57.11	28/01/2015	16/02/2015
KPU076	116042.07	194492.45	-1267.95	140.80	43.04	-22.46	02/02/2015	07/02/2015
KPU077	116236.74	194259.06	-1284.41	500.80	281.45	-52.40	08/02/2015	23/02/2015
KPU078	116042.47	194491.79	-1268.31	245.80	52.60	-51.96	18/02/2015	26/03/2015
KPU079	116185.99	194234.43	-1277.91	719.80	316.75	-50.71	26/02/2015	12/03/2015
KPU080	116037.35	194546.21	-1268.05	311.90	24.30	-80.48	13/03/2015	20/03/2015
KPU081	116185.46	194234.06	-1278.16	632.80	304.66	-55.01	20/03/2015	04/04/2015
KPU082	116185.29	194234.30	-1277.96	482.60	307.00	-44.06	04/04/2015	13/04/2015
KPU083	116184.66	194234.76	-1278.29	383.30	306.73	-33.91	14/05/2015	22/04/2015
KPU084	116036.43	194510.81	-1267.65	361.30	340.18	-75.79	04/05/2015	11/05/2015
KPU085	116028.72	194529.93	-1266.93	251.70	120.83	-50.98	11/05/2015	17/05/2015
KPU086	116038.23	194543.90	-1267.67	221.70	125.50	-51.00	17/05/2015	20/06/2015
KPU087	116040.11	194553.15	-1267.05	192.00	112.10	-45.40	21/05/2015	25/05/2015
KPU088	116185.53	194233.15	-1278.09	27.00	0.00	0.00	25/05/2015	27/05/2015
KPU089	116185.53	194233.15	-1278.09	642.00	301.60	-63.80	26/05/2015	09/06/2015
KPU090	116192.73	194226.94	-1278.93	551.80	312.52	-47.92	23/07/2015	14/07/2015
KPU091	116185.55	194233.83	-1278.07	775.80	300.50	-58.77	23/06/2015	14/07/2015
KPU092	116185.63	194234.11	-1278.01	633.00	307.47	-52.63	15/07/2015	30/07/2015
KPU093	116234.80	194252.70	-1283.16	488.10	311.00	-43.00	31/07/2015	11/08/2015
KPU093W1	116234.80	194252.70	-1283.16	1001.10	315.50	-46.50	12/08/2015	10/09/2015
KPU094	116176.75	194235.40	-1276.23	257.80	296.02	-22.47	08/10/2015	14/10/2015
KPU095	116176.77	194235.70	-1276.66	551.10	302.46	-48.43	15/10/2015	26/10/2015
KPU096	116190.20	194229.10	-1278.27	425.60	297.38	-41.60	29/09/2015	07/10/2015
KPU097	116190.57	194228.90	-1278.07	452.80	297.66	-49.84	18/09/2015	28/09/2015

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 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 of 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 of 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, mineralization, and structure was done on standardised paper templates and then captured and validated on import into MS Access. From hole KPU015 onwards, all logging was done directly into MS Access. All geotechnical logging was done directly into MS Access.

All drill cores were photographed both wet and dry prior to sampling.

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.

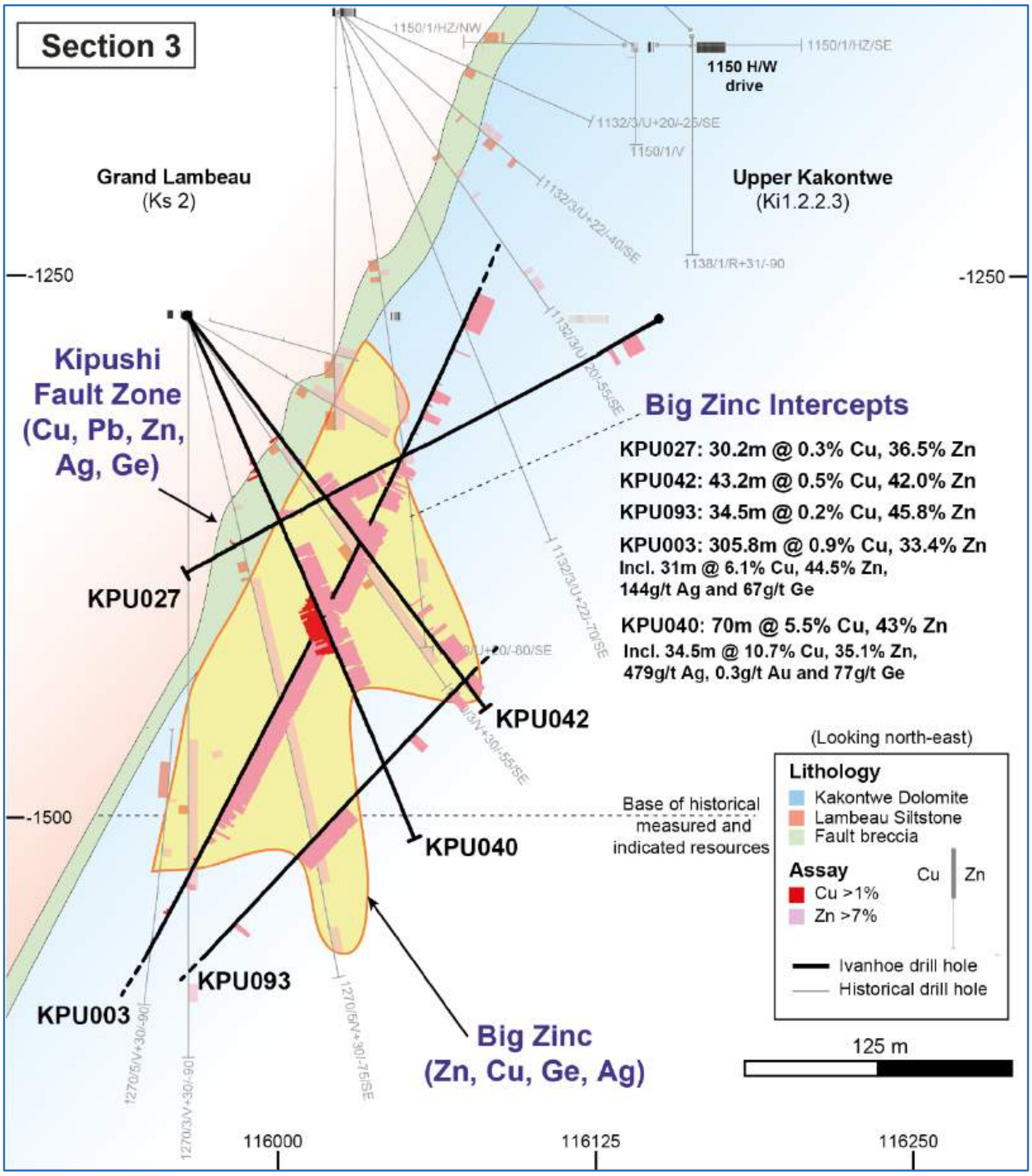
10.2.6 Results

Drilling has confirmed that zinc and copper mineralization extends below the extent of the Techpro historical estimate to 1,810 mRL with the deepest intersection recorded in hole KPU079.

In addition to confirming substantial widths and zinc grades within the Big Zinc zone, some of the KICO holes also intersected zones of high-grade copper and precious metals within the Big Zinc zone, e.g. drillhole KPU040 which returned 34.5 m grading 35.1% Zn, 10.7% Cu, 479 g/t Ag, 77 g/t Ge, and 0.30 g/t Au.

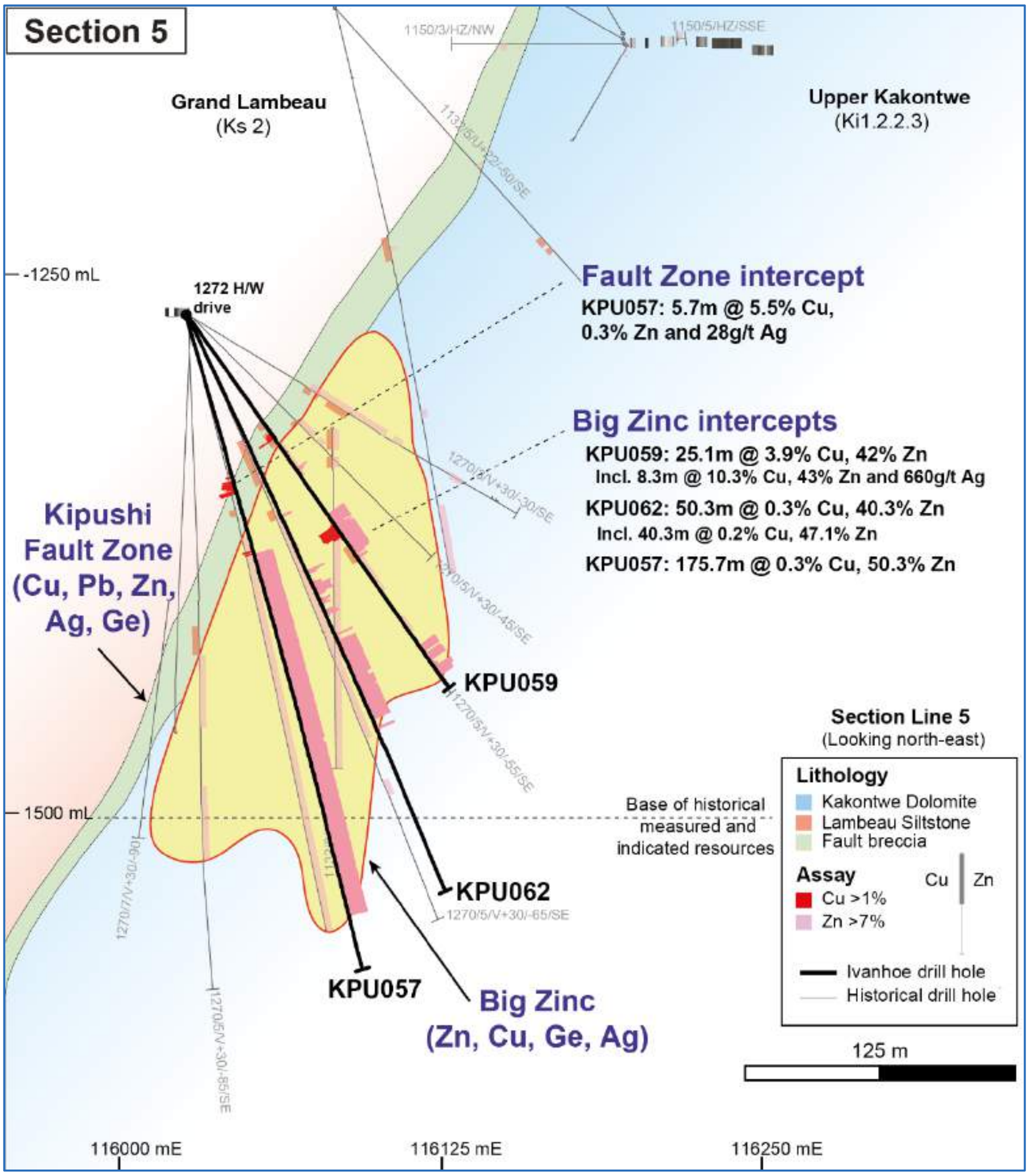
Figure 10.2 to Figure 10.6 show schematic sections illustrating the KICO drilling results within the Big Zinc zone and Fault Zone. The geometry of the Big Zinc and copper-rich and zinc-rich mineralized zones at depth below the Big Zinc zone are shown schematically in Figure 10.7.

Figure 10.2 Schematic Drill Section 3 Showing Drillholes through the Big Zinc Zone



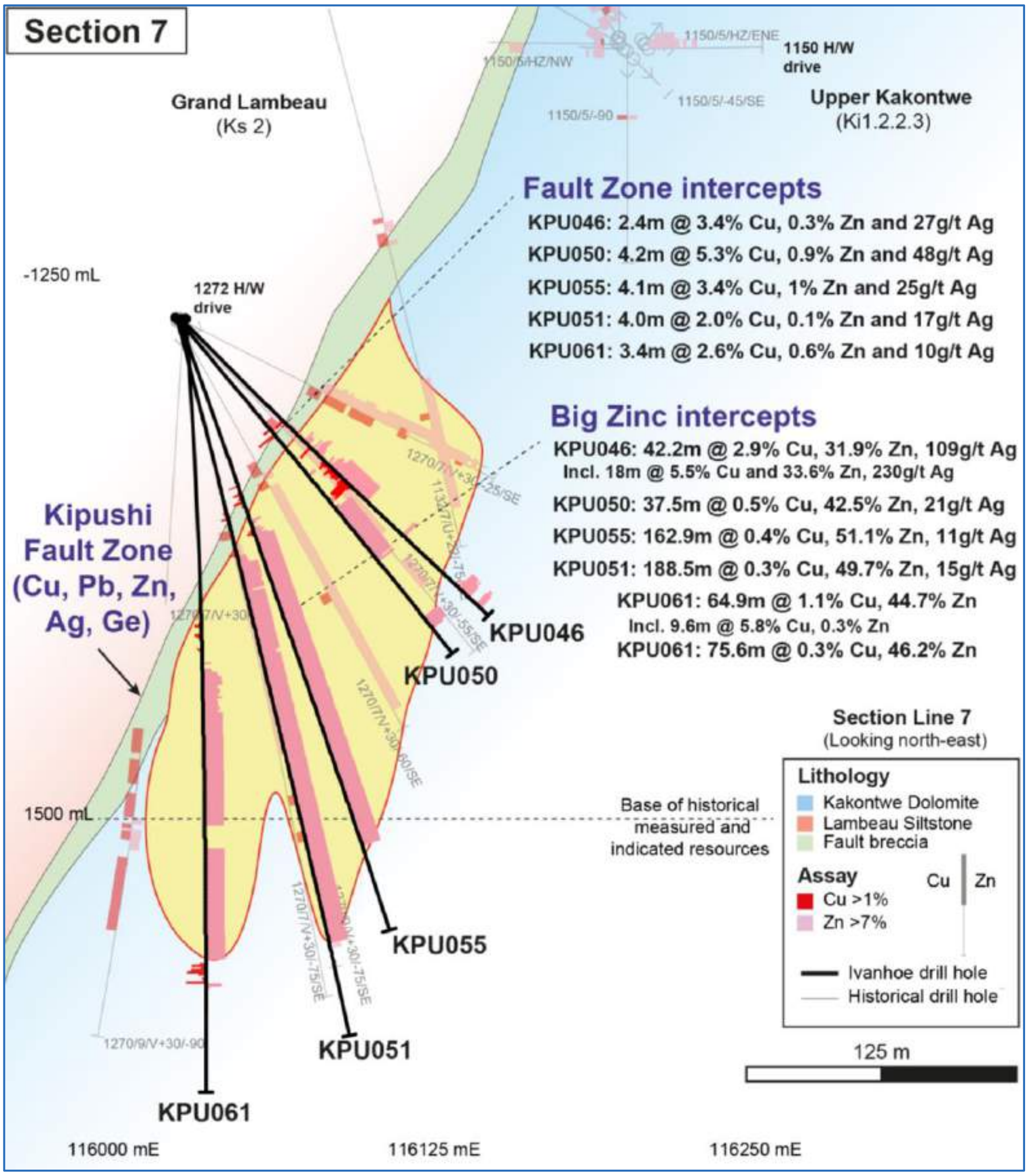
Source: Ivanhoe Mines (2015)

Figure 10.3 Schematic Drill Section 5 Showing Drillholes through the Big Zinc Zone



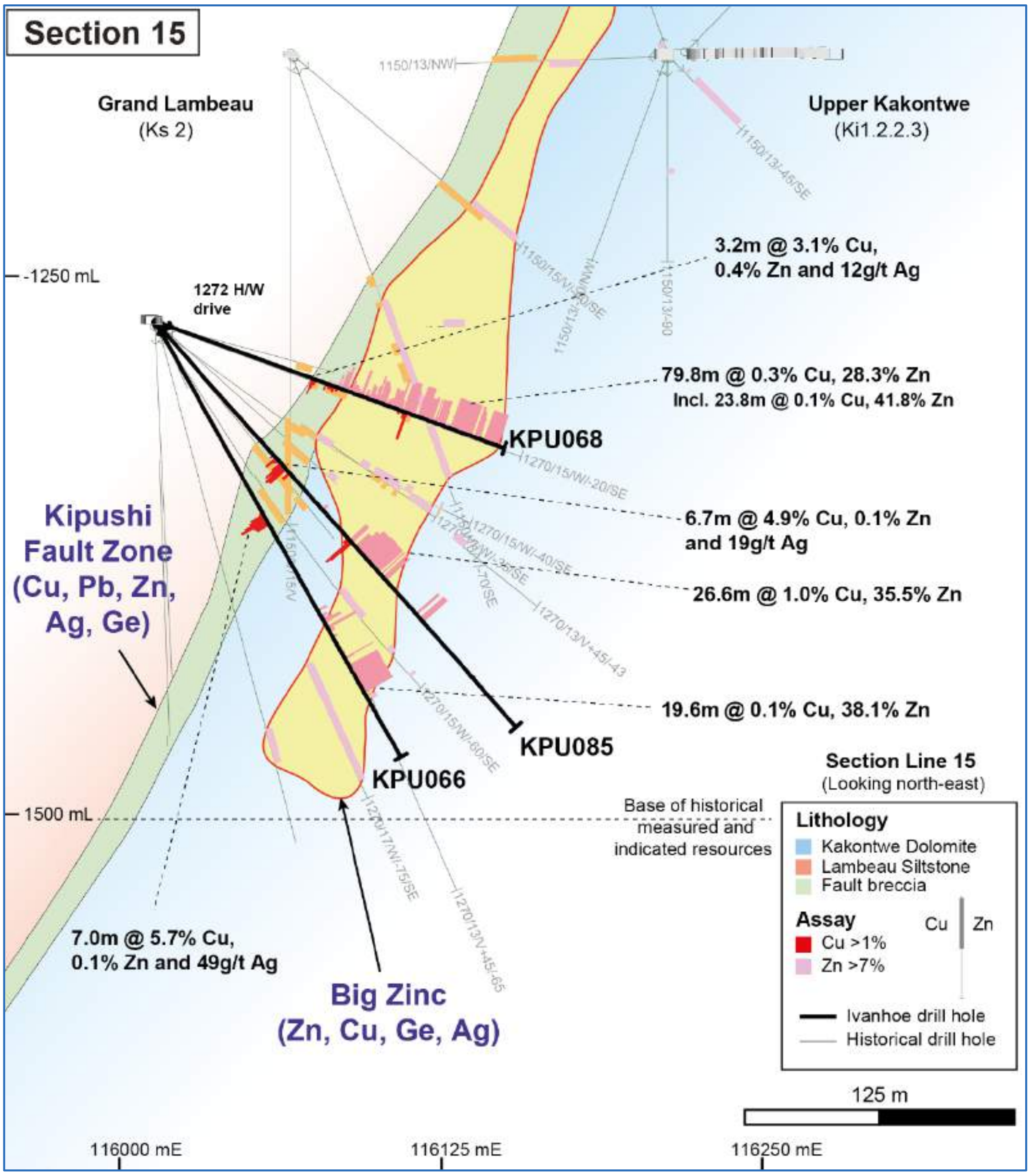
Source: Ivanhoe Mines (2015)

Figure 10.4 Schematic Drill Section 7 Showing Drillholes through the Big Zinc Zone



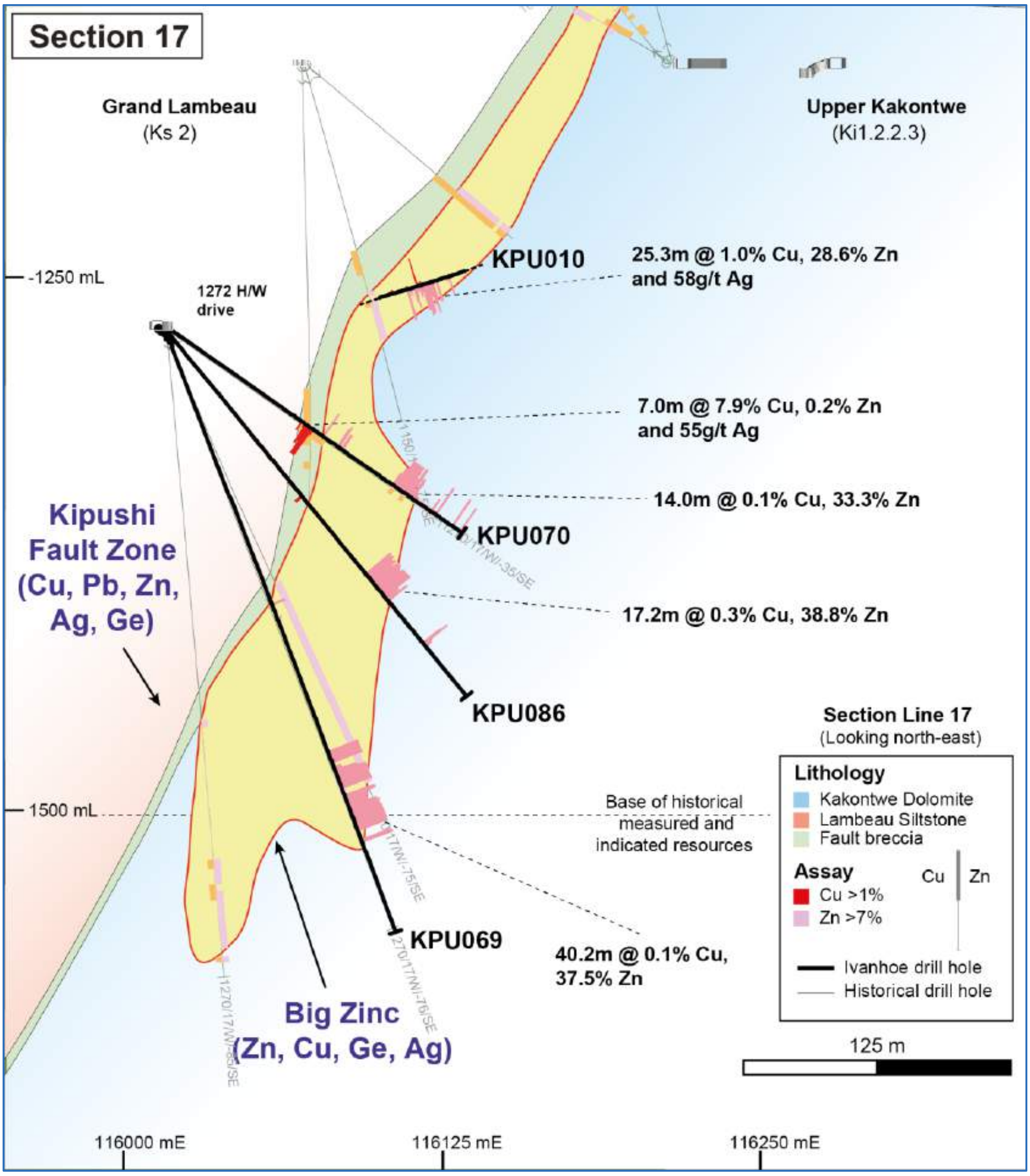
Source: Ivanhoe Mines (2015)

Figure 10.5 Schematic Drill Section 15 Showing Drillholes through the Big Zinc Zone



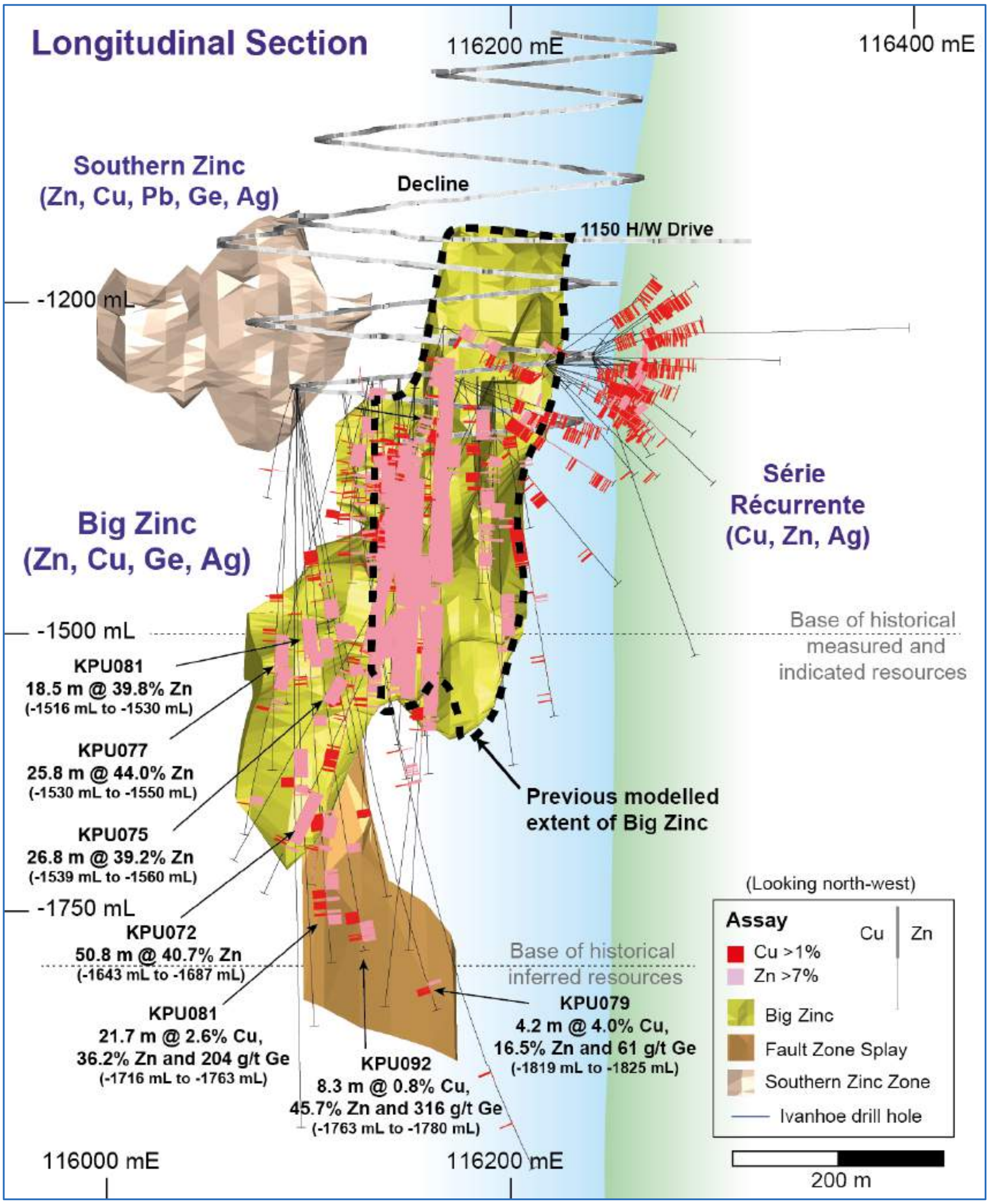
Source: Ivanhoe Mines (2015)

Figure 10.6 Schematic Drill Section 17 Showing Drillholes through the Big Zinc Zone



Source: Ivanhoe Mines (2015)

Figure 10.7 Schematic Drill Section Looking North–west Showing Drillholes through the Big Zinc Zone, and Showing Additional Intersections at Depth

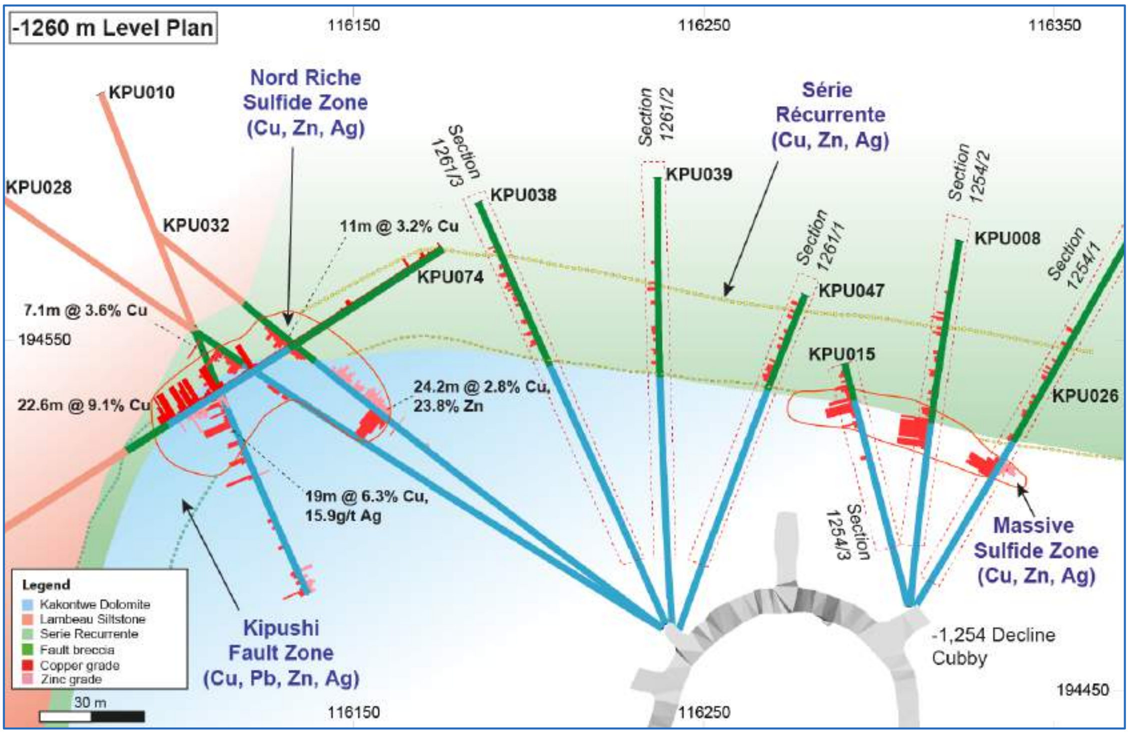


Source: Ivanhoe Mines (2015)

A plan projection of KICO drilling in the Copper Nord Riche and Série Récurrente zones is shown in Figure 10.8. 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 sulphides (chalcopyrite and sphalerite) as shown in section in Figure 7.17.

The Série Récurrente zone contains a westerly-plunging lense of high-grade copper-rich massive sulphide that extends from the Série Récurrente zone into the Upper Kakontwe. Drilling by Gécamines intersected this zone up-plunge but it was not mined.

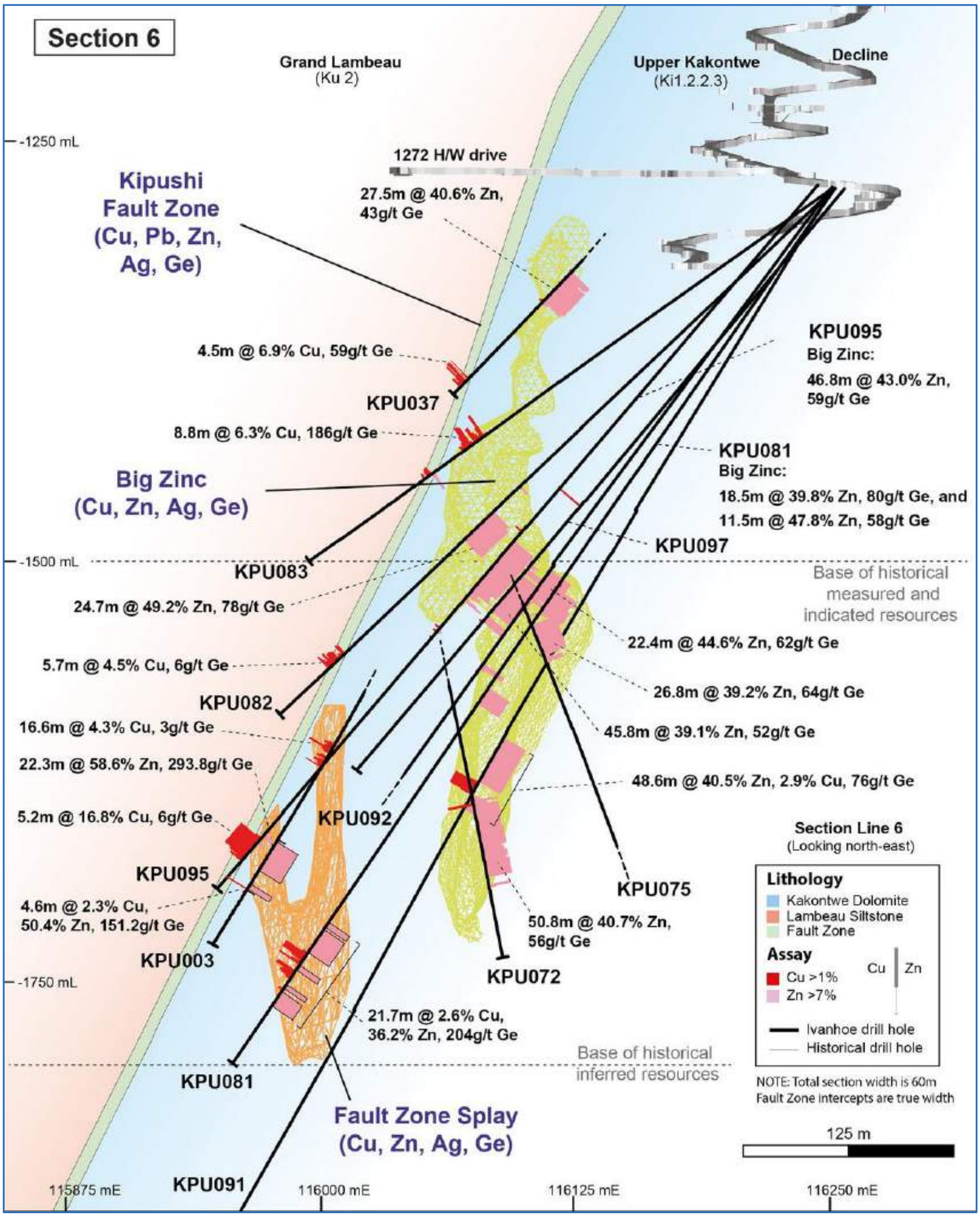
Figure 10.8 Drill Plan of 1,260 mRL Showing KICO Drilling in the Copper Nord Riche and Série Récurrente Zones



Source: Ivanhoe Mines (2015)

In addition to confirming substantial widths and zinc grades within the Big Zinc zone, some of the KICO holes have also intersected zones of high-grade copper and precious metals within the Big Zinc zone. A high grade massive sulphide lense within the Série Récurrenté zone and a germanium-rich zone that occurs as a splay off the Fault Zone at depth have also been defined (Figure 10.9).

Figure 10.9 Schematic Drill Section 6 Showing Drillholes through the Big Zinc Zone and the Fault Zone Splay



Source: Ivanhoe Mines (2015)

10.2.7 QP Comment

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

- Core recoveries are typically excellent,
- Drillhole orientations are mostly appropriate for the mineralization styles at Kipushi and adequately cover the geometry of the various mineralized 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

11.1 Gécamines Sampling Approach

Sampling by Gécamines was selective and lower grade portions of the mineralized intersections were not always sampled. Drill cores had a diameter of between 30 and 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 mineralization 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 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.
- Sulphur 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 crushers

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

11.3 KICO Sample Preparation Methods

All sample preparation, analyses and security measures were carried out under standard operating procedures set up by KICO for the Kipushi Zn-Cu Project. These procedures have been examined by the 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 mineralization 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 mineralization 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 mineralized zones.

Sample preparation was completed by staff from KICO and its affiliated companies at its own internal containerised laboratories at Kolwezi and Kamoa (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 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 g to 1000 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 pulverized test, and 10 g of wet material was used for the wet pulverized test.

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; and
- A 90 g sample for office archives.

Figure 11.2 Containerised Sample Preparation Facility at the Kolwezi Laboratory

A Drying oven and sample racks

B Crusher and pulveriser

C Compressor and sample trays

D Coarse quartz blank material used for flushing between samples

Figure 11.3 Sample Preparation Facility at the Kamoa Laboratory

A Drying oven

B Crusher and riffle splitter

C Crushers

D Labtech Essa LM2 pulverisers

E Dust filtration system

11.4 KICO Analytical Approach

The laboratory analytical approach and suite of elements to characterize the major and trace element geochemistry of the Big Zinc zone 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 mineralization styles from Gécamines drillholes which intersected the Fault Zone and Big Zinc zone.

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 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; and
- Au, Pt, and Pd by 10 g (due to inherent high sulphur content of the samples) lead collection fire assay with an ICP-OES finish.

For 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

11.5 Quality Assurance and Quality Control

11.5.1 QAQC Approach

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

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

QAQC 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 QAQC programme monitored:

- Sample preparation screen test data,
- Analytical data obtained from certified reference materials (CRM), blanks (BLK), and crushed duplicates (CRD), and
- Internal laboratory pulverized 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 are stored in an MS Access database. QAQC data were exported from the Access database into software applications for creating monitoring charts and comparison charts. The number of samples reviewed by Sketchley (2015a) comprised 9,887 routine samples, 502 CRMs, 434 blank samples, 514 crushed duplicates and 812 laboratory duplicates.

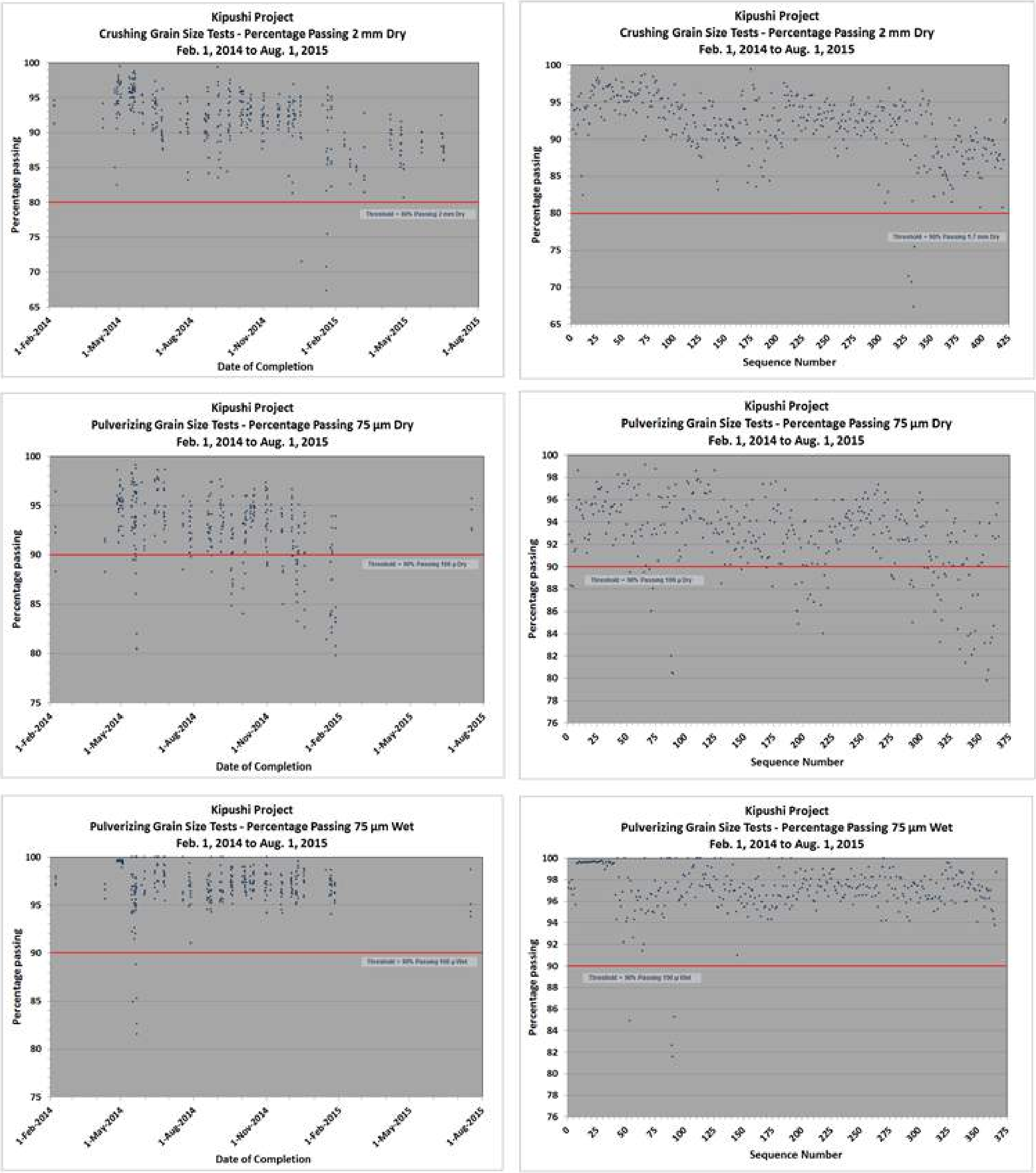
All of the sample batches submitted to BVM had approximately 5% CRMs, 5% blanks, and 5% crushed reject duplicates inserted into the sample stream.

11.5.2 Laboratory Performance

11.5.2.1 Sample Preparation

Final statistical charts illustrating results from the Kolwezi and Kamoa sample preparation laboratories grain size monitoring are presented in Figure 11.6. The majority of samples pass 80% dry for the crushing step. For the pulverizing step, almost all samples pass 90% wet and the majority of samples pass 80% dry. The results are acceptable for styles of mineralization with low heterogeneity.

Figure 11.5 Crushing and Pulverising Grain Size Monitoring Charts



Source: Sketchley (2015a)

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 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 program 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. Where CRM failures were identified, re-assays were requested on the failed CRM together with several adjacent routine samples. Re-assay results were assessed in the same manner. The results of the CRM programme for the main elements of economic interest are shown in Table 11.3.

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	–	2604	–	4.54	–
Oreas 163	–	1.71	–	–	–	–	–	9.98	–	–	–	–	–
Oreas 165	–	10.20	–	–	–	–	–	8.28	–	–	2485	–	–
Oreas 166	–	8.75	–	10.80	–	–	–	11.29	–	–	2077	–	–
Kip 1	57.57	–	–	21.20	–	88.0	26	34.06	908	3254	–	–	–
Kip 2	25.01	–	–	–	165.0	49.3	96	24.07	1401	1548	–	–	0.188
Kip 3	–	5.78	–	36.00	–	–	–	6.10	1431	–	–	–	0.875
Kip 4	5.00	5.24	–	22.20	–	11.5	51	17.00	2327	–	–	–	–

Notes: AR = Aqua Regia; FA = Fire Assay

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 4 positive failures, and AMIS 153 had 3 positive failures. Three of the 4 failures for AMIS 147 and 2 of the 3 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 negative bias due to the partial digest of the method.
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 3 CRMs is poor.	KIP 1 displays no failures despite a strong negative bias of almost 11%, as a result of wide tolerance limits. The single KIP 2 result is a marginal negative failure. KIP 4 displays one positive failure and poor precision due to the low value.
Au (FA)	Accuracy and precision for all CRMs tends to be poor.	AMIS 147 displays 2 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.
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.

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

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 144, and AMIS 149 were excluded from the QAQC review as they were used only once each.

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.

11.5.2.3 Blanks

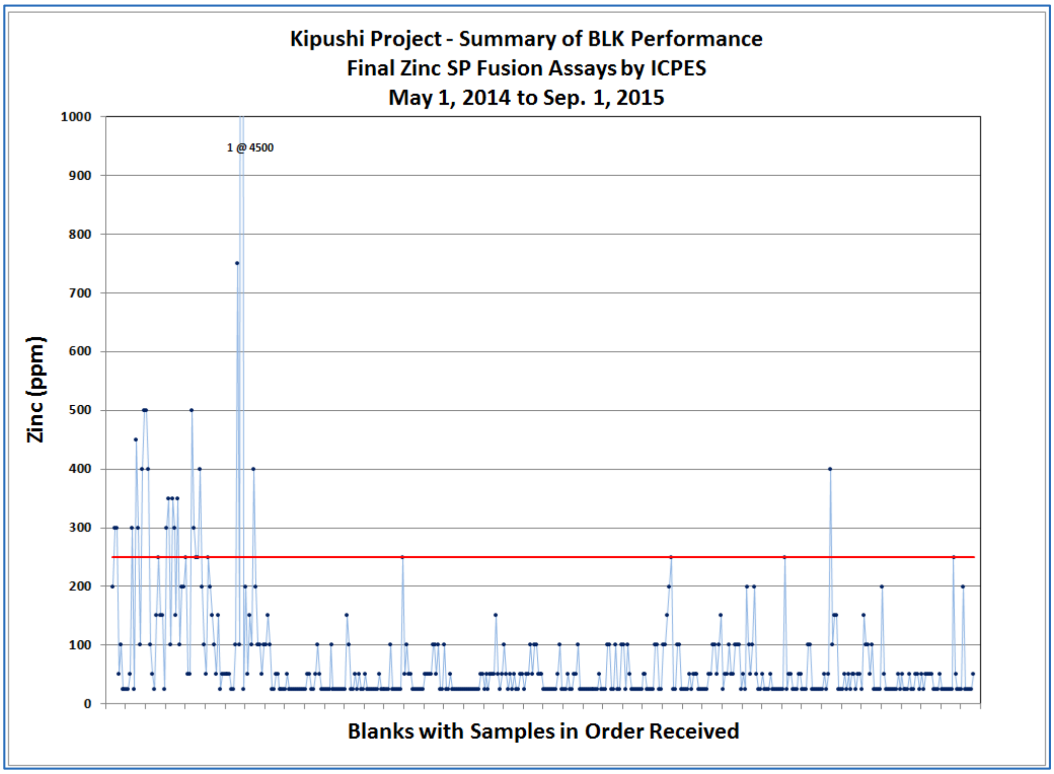
Locally obtained barren coarse quartz vein material was used to monitor contamination and sample mix-ups (Figure 11.2). This material was previously analysed in separate programmes (both Kipushi re-sampling and Kamoa programmes) to ensure that it was barren of the elements of interest. Analytical performance 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.

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 11.6. 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 2000 ppb. The anomalous values may be related to spurious gold within the quartz vein material.

Figure 11.6 Blank Sample Performance for Zinc by Sodium Peroxide Fusion



Source: Sketchley (2015a)

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 pulverizing 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.

Pulverized duplicates were routinely done by BVM during assaying and were used to monitor the combined precision of the pulverizing 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 4–12% for all elements except gold and rhenium, which are 34% and 19% respectively.

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), germanium, sulphur, cadmium and cobalt 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, arsenic, 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, the differences decrease to less than 20% for CRDs and 10% for LREPS. Larger differences for silver (peroxide fusion), gold and rhenium 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% to 20% for crushed duplicates, and better than 5% to 10% for pulverized duplicates.

Asymptotic precision values for CRDs and LREPs are 10% or below for all elements, except gold and rhenium, which have a level of 19% for CRDs and 13–22% for LREPs. All elements tend to have better precision for pulverised duplicates than crushed duplicates, as expected. Similarly, the practical detection limit for pulverized 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. 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 QAQC protocol implemented by KICO concluded the following:

- The results of the QAQC 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-emphasized.
- 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 QAQC routines implemented for the Gécamines samples and 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 mineralized 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 program 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 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 QAQC data, geological logging and specific gravity information, and downhole and collar coordinate surveys.

12 DATA VERIFICATION

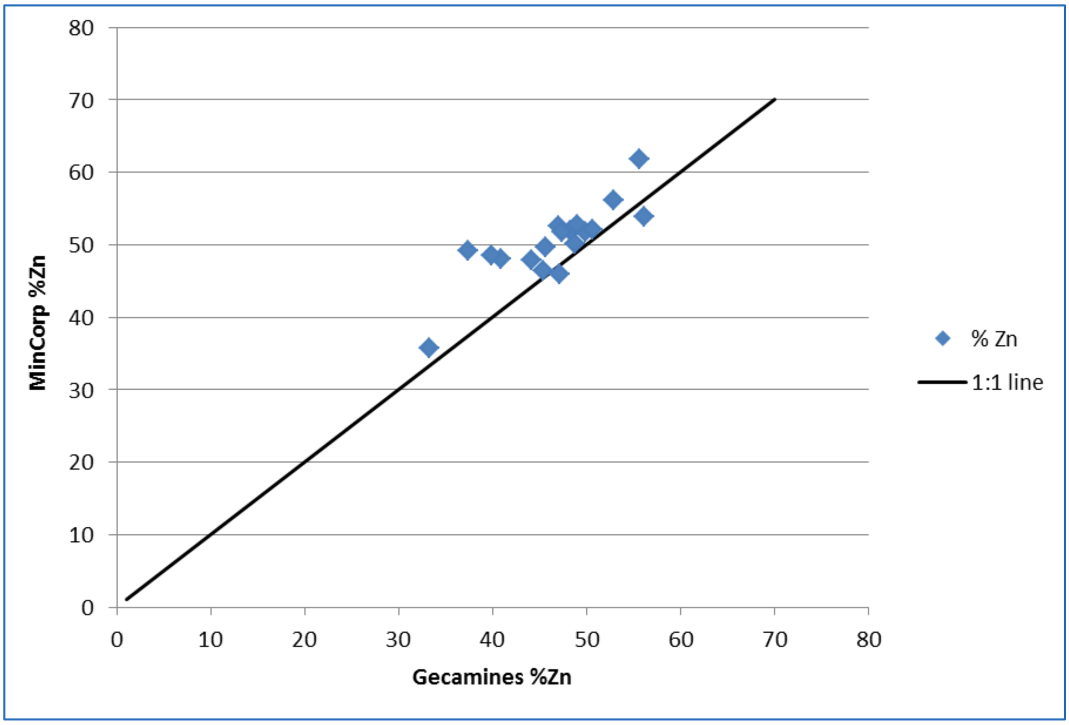
A comprehensive re-sampling programme was undertaken on historical Gécamines drillhole core from the Big Zinc zone and Fault Zone below 1,270 mRL at the Kipushi Mine. The objective of the exercise was to verify historical assay results and to assess confidence in the historical assay database for its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

In addition, 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 twenty 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



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 February to 22 February 2013 by the QP, Mr Robertson, in order to view the condition of the existing Gécamines drillhole cores from holes collared on the 1,270 mRL, 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 have composite assay data results and individual sample assays are not available or have not been captured.

Holes were selected to cover the various mineralization 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 range from -25˚ to -75˚ to cover the dip extent of the mineralization. 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 zone.

Re-sampling of the drill core was supervised by the MSA QP in a follow-up site visit from 22 April to 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
1270	3	-55; -75	-75	31	Medium Cu zone in Fault Zone; wide intersection though Big Zinc zone, although not true thickness
1270	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.
1270	7	-55; -75	-25	21	Thick high Cu zone in Fault Zone; intersects upper part of Big Zinc zone
1270	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)
1270	11	-45; -65	-25	15	Intersects upper part of Big Zinc zone; includes narrow zones of high Cu
1270	13	-65	-75	19	Narrow zones of high Cu; intersects lower part of Big Zinc zone
1270	15	-20	-40	12	High Cu in Fault Zone; intersects middle zone of Big Zinc zone
1270	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 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 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 QAQC

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 mineralized intersections, at a cut-off of 7% Zn, between historical and re-sampling results is shown in Table 12.3. The comparison revealed an 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 are excluded, 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 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 mineralization.
- Disregarding the few outliers, BVM slightly under-reports lead compared to Gécamines.
- Sulphur 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 Mineralized Intersections between Gécamines and the re-Sampling Programme using a cut-off of 7% Zn

	Gécamines data						Re-Sampling programme					
Hole_ID	**From**	**To**	**Interval[2]**	**Zn %**	**Cu %**	**Calculated Density**	**From**	**To**	**Interval[2]**	**Zn %**	**Cu %**	**Density[3]**
1270/3/V+30/-75/SE[1]	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

Note:

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

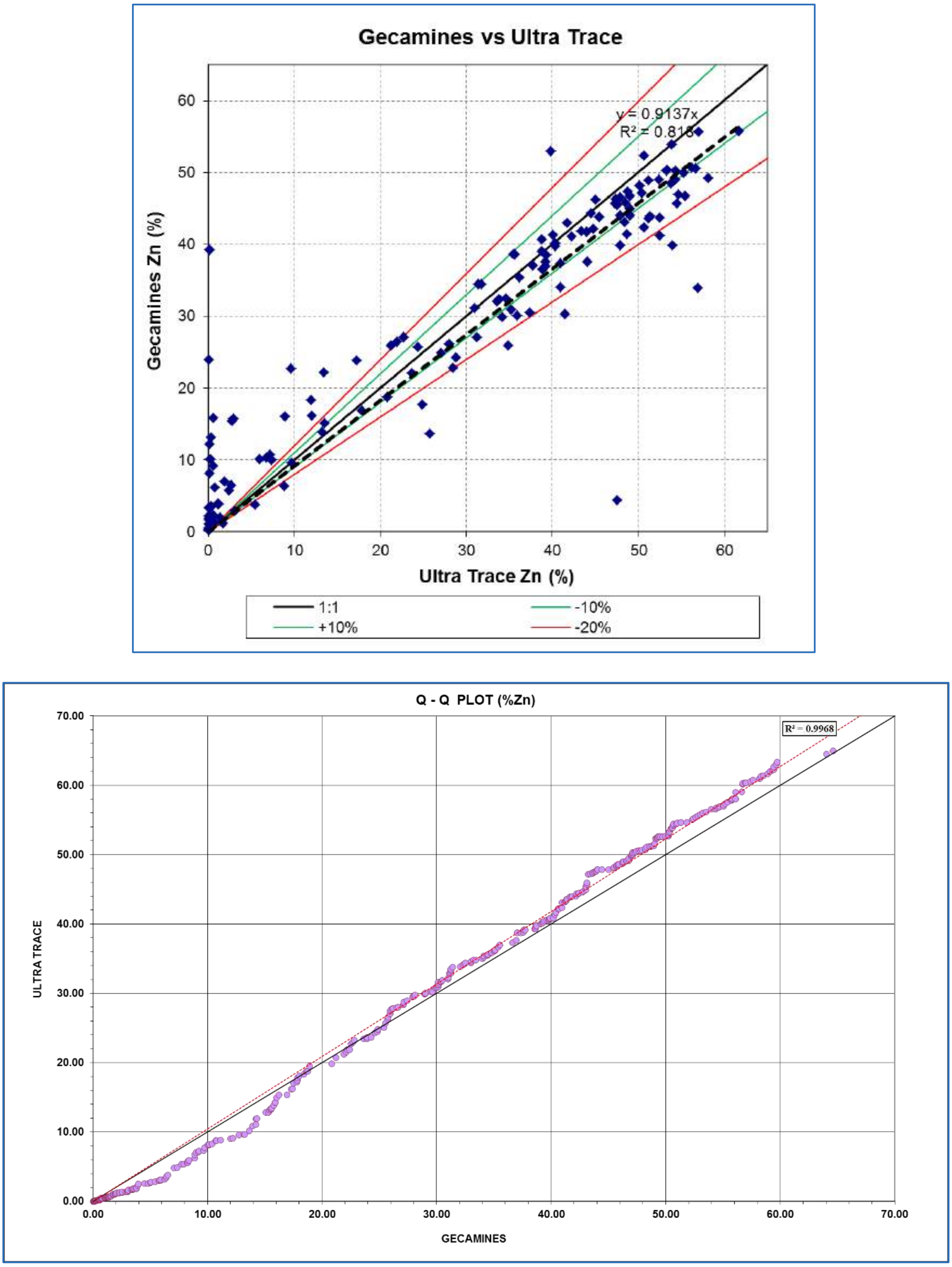
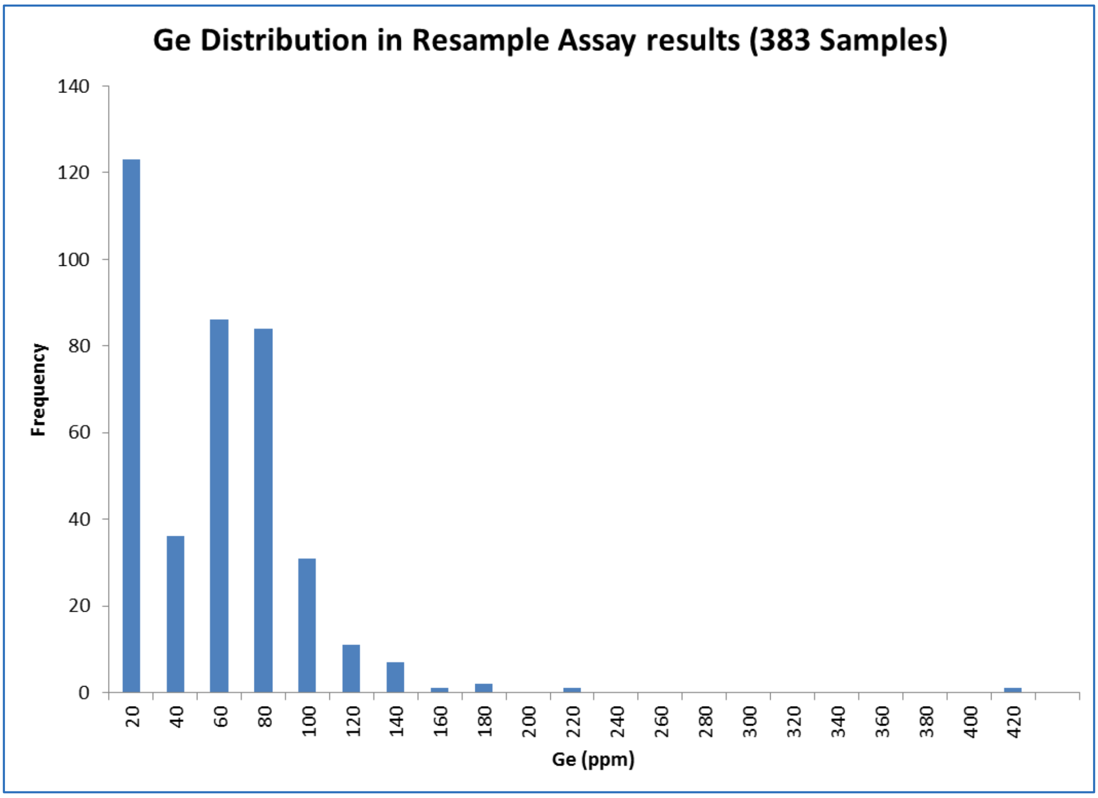


Figure 12.3 Histogram Plot of BVM Ge Results



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.

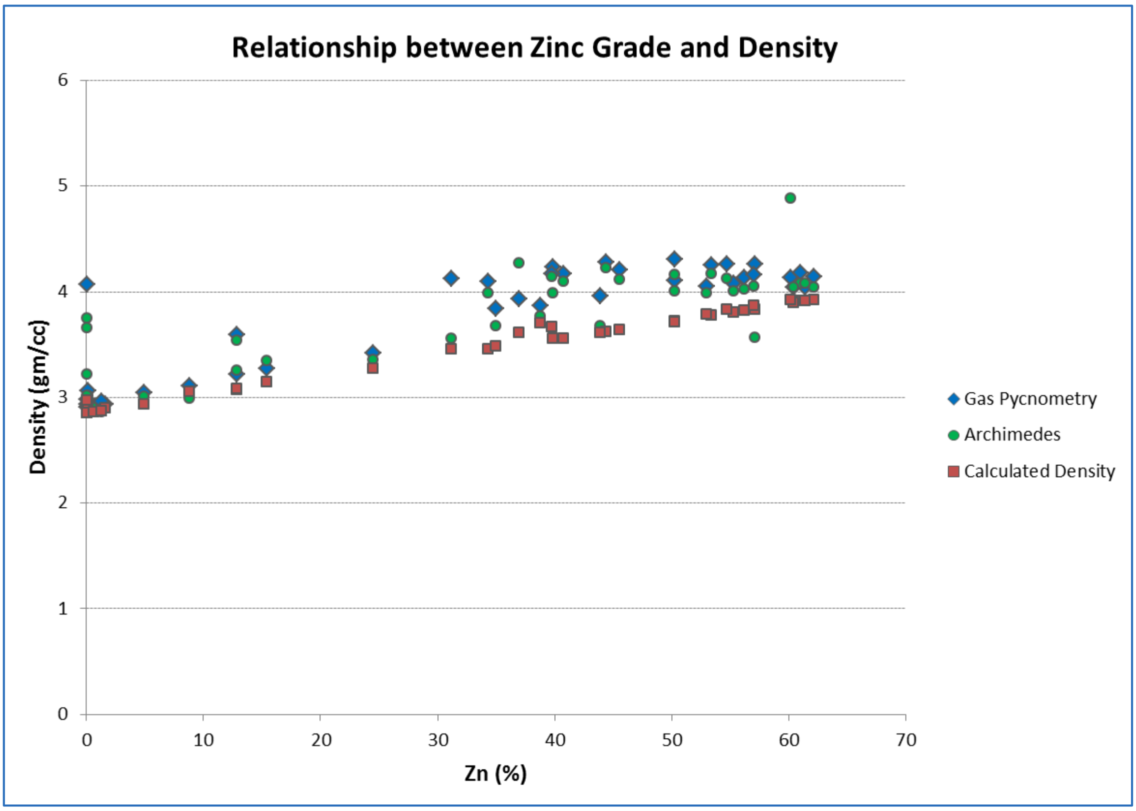
Gécamines used the following formula, derived mainly for the Fault Zone, to calculate density for use in its tonnage estimates:

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

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



12.3 Re-logging Programme

KICO geologists undertook remarking and re-logging of all the available Gécamines drillholes that intersected the Big Zinc zone, 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 mineralization and complete sampling of the KICO twin holes allowed for determining the limits of mineralization (Figure 12.9).

The KICO drillholes were more completely sampled in lower grade mineralization 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 mineralization. More detail and grade resolution in therefore observed in the KICO sampling compared to Gécamines sampling where sample lengths were based on homogenous zones of mineralization 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 & 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

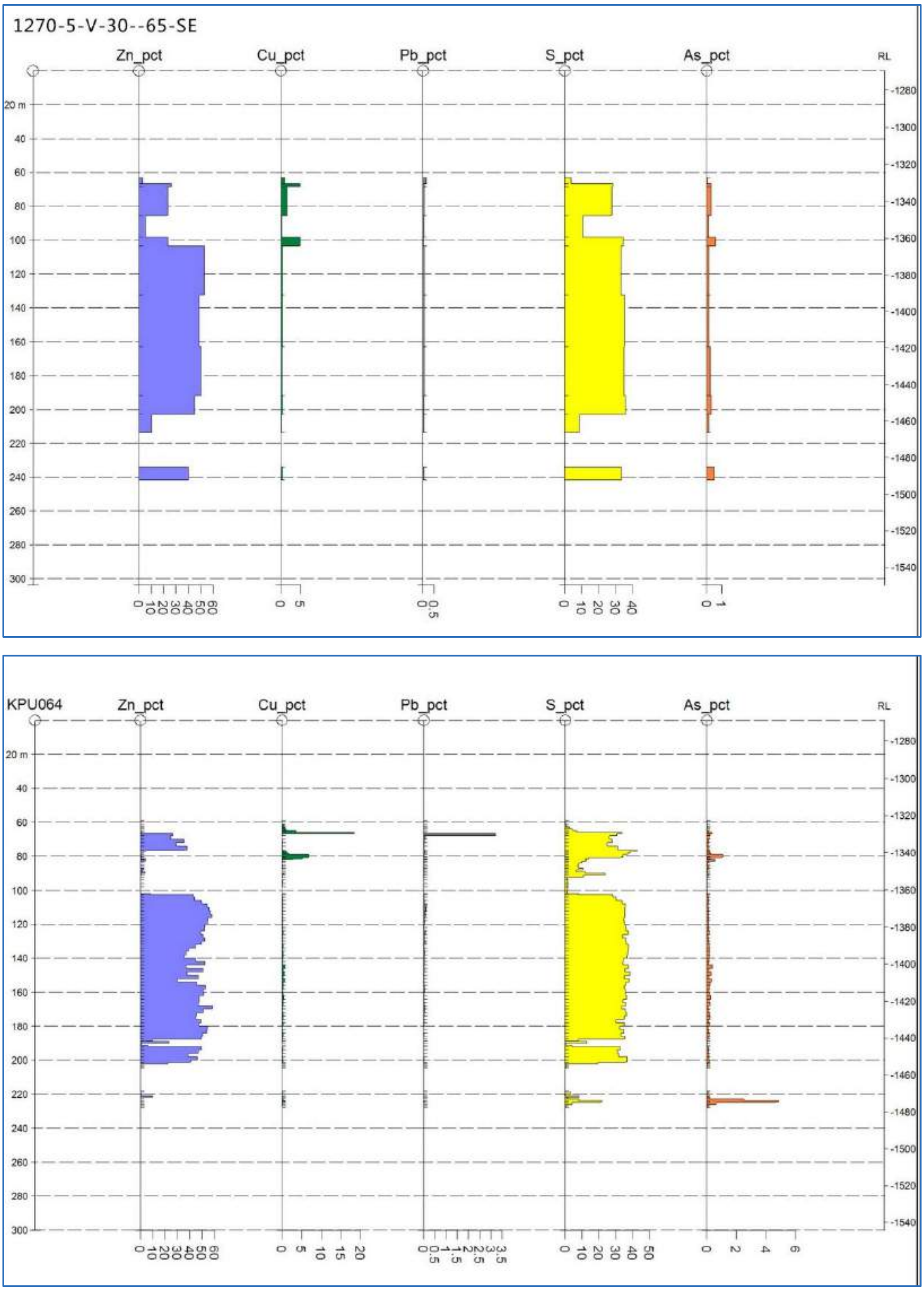


Figure 12.6 Comparison between Gécamines hole 1270/5/V+30/-55/SE and KICO Twin Hole KPU059

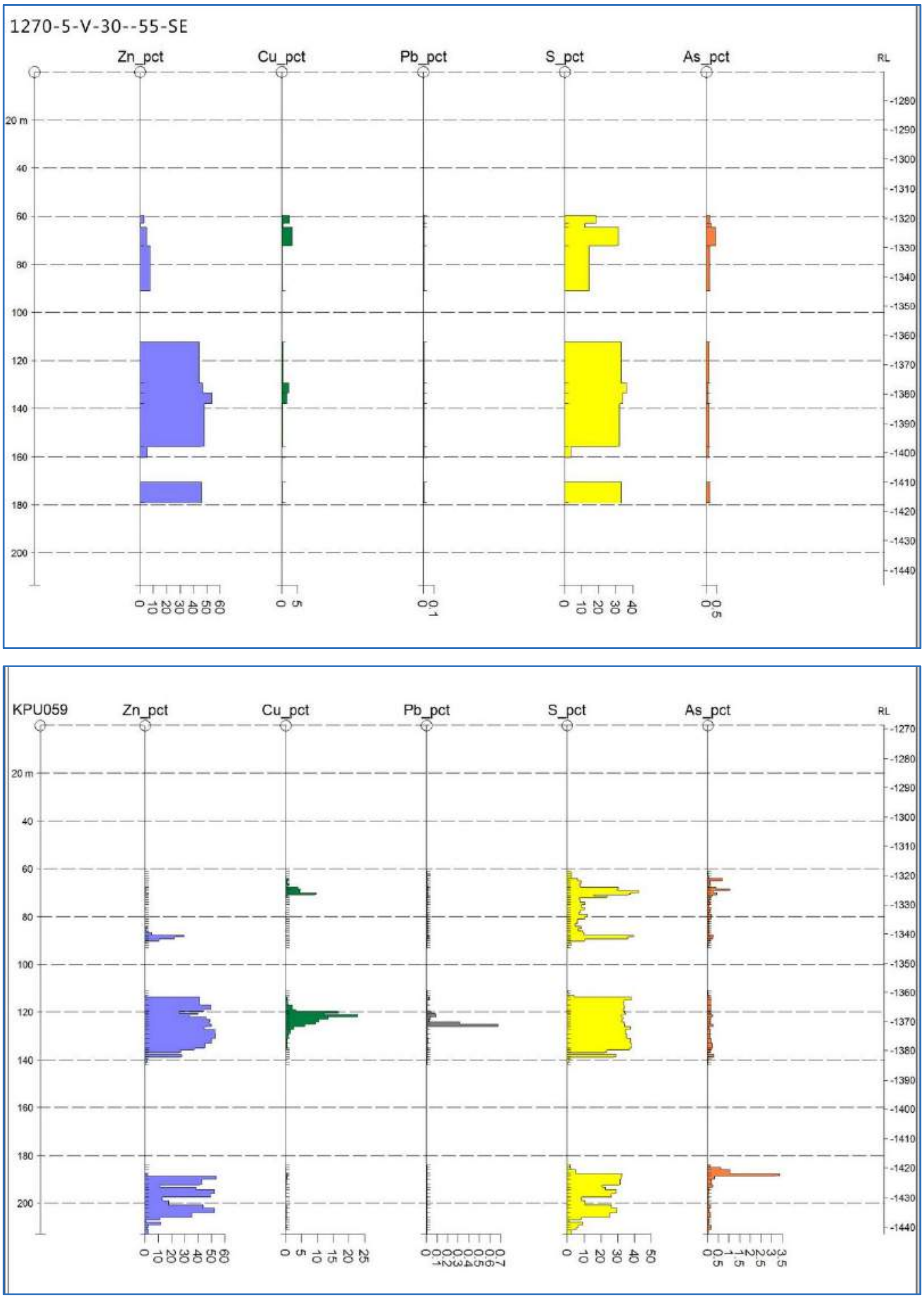


Figure 12.7 Comparison between Gécamines hole 1270/17/W/-76/SE and KICO Twin Hole KPU069

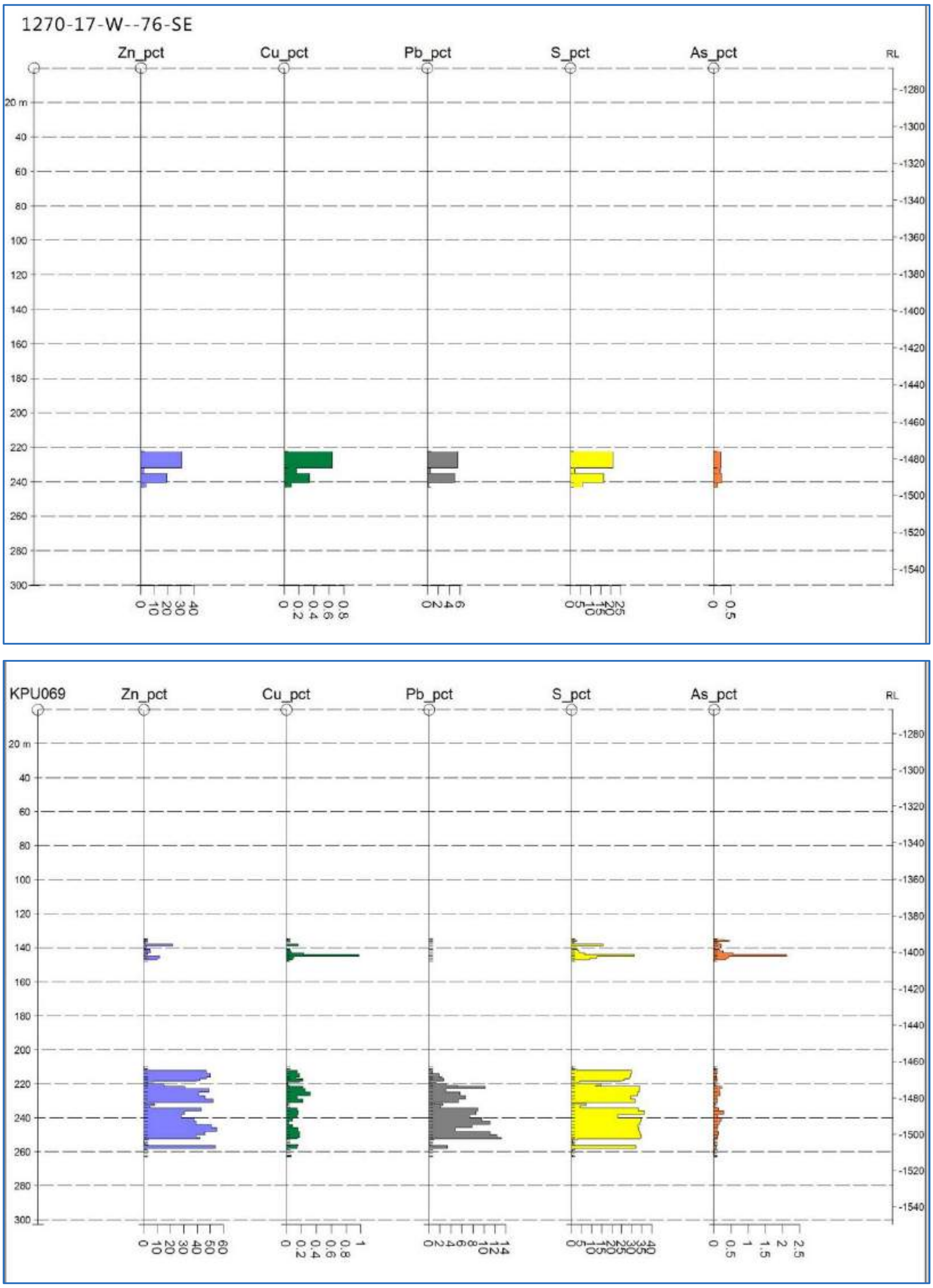


Figure 12.8 Comparison between Gécamines hole 1270/15/W/-20/SE and KICO Twin Hole KPU068

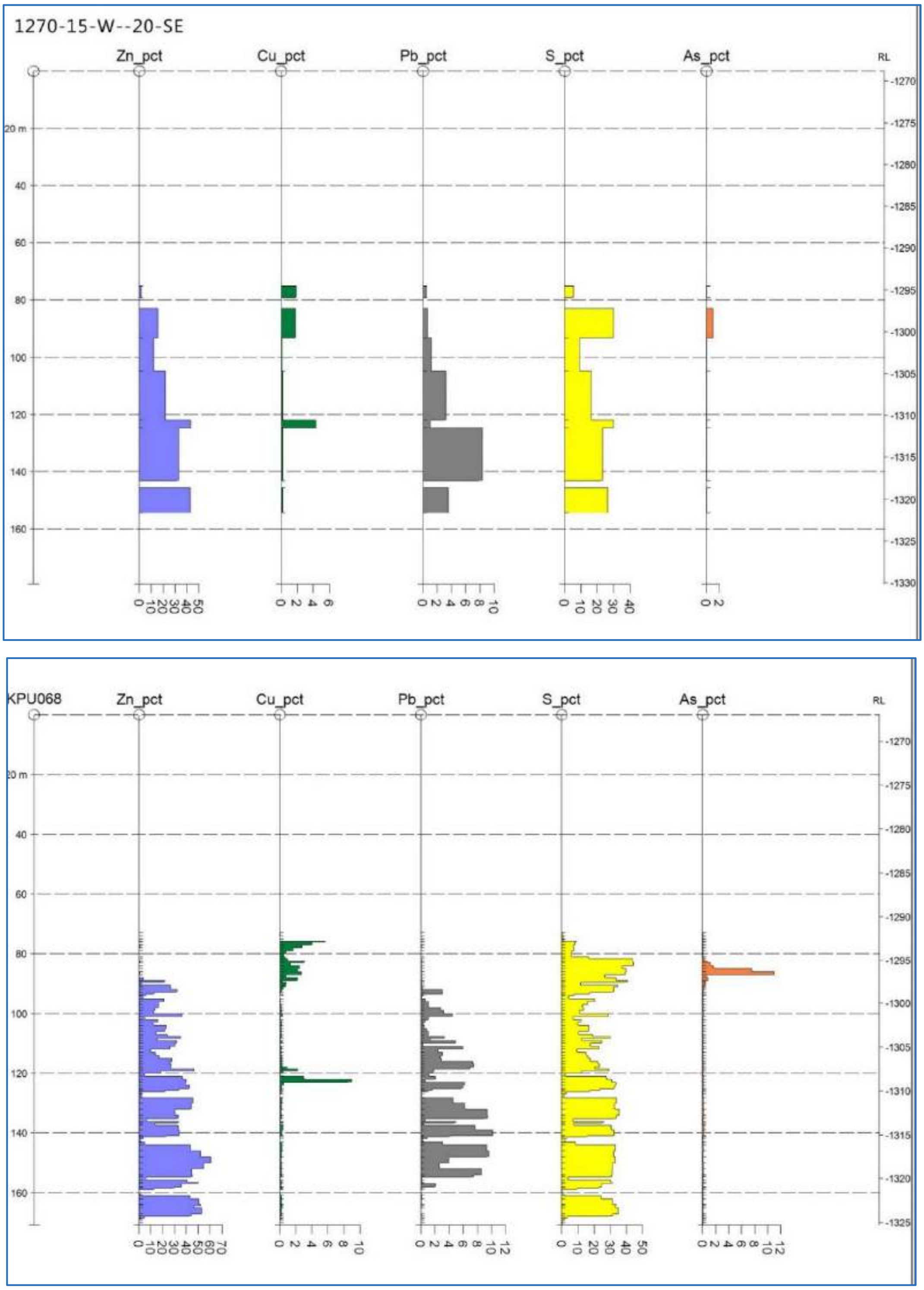


Figure 12.9 Comparison between Gécamines hole 1270/7/V+30/-75/SE and KICO Twin Hole KPU051

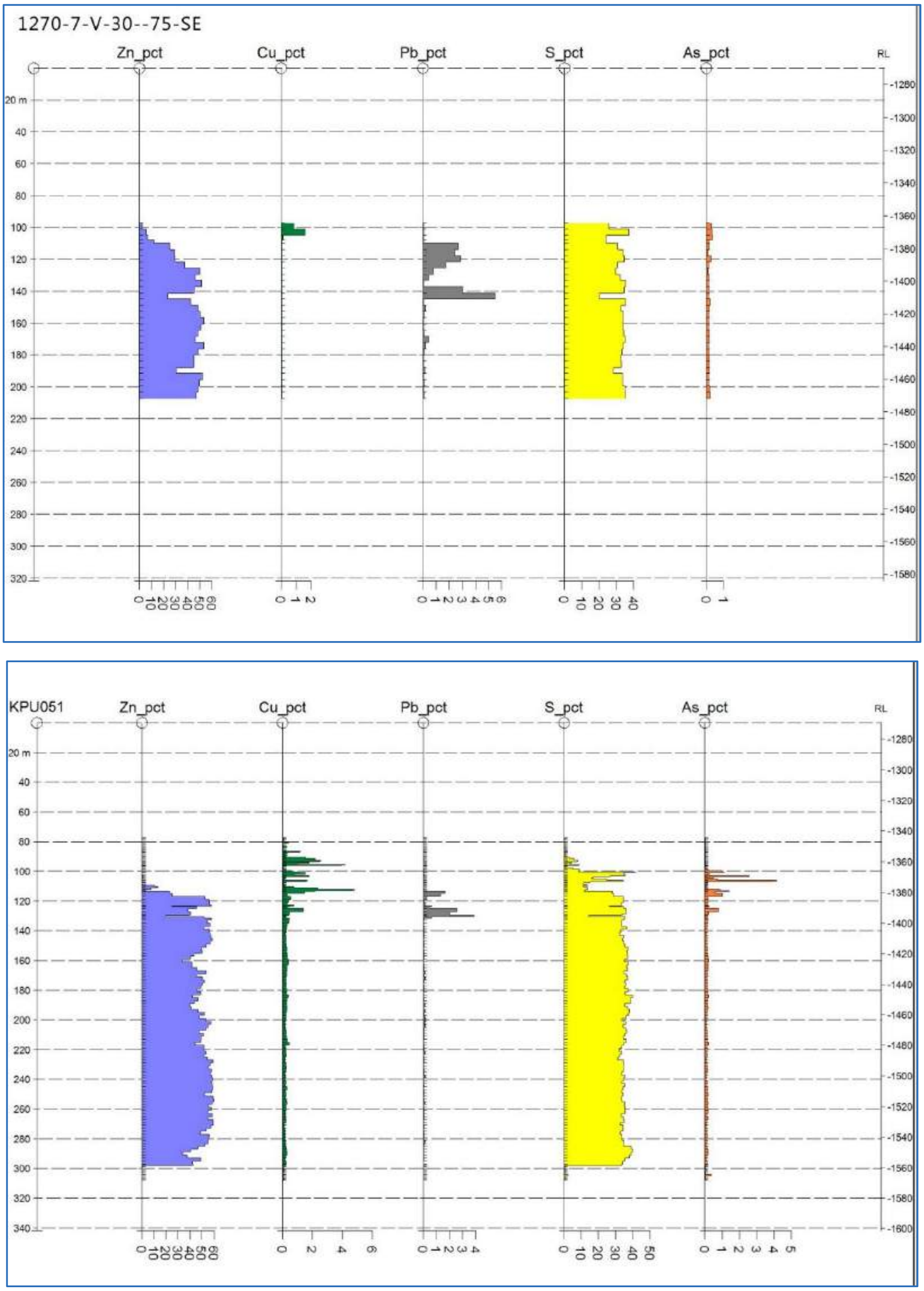
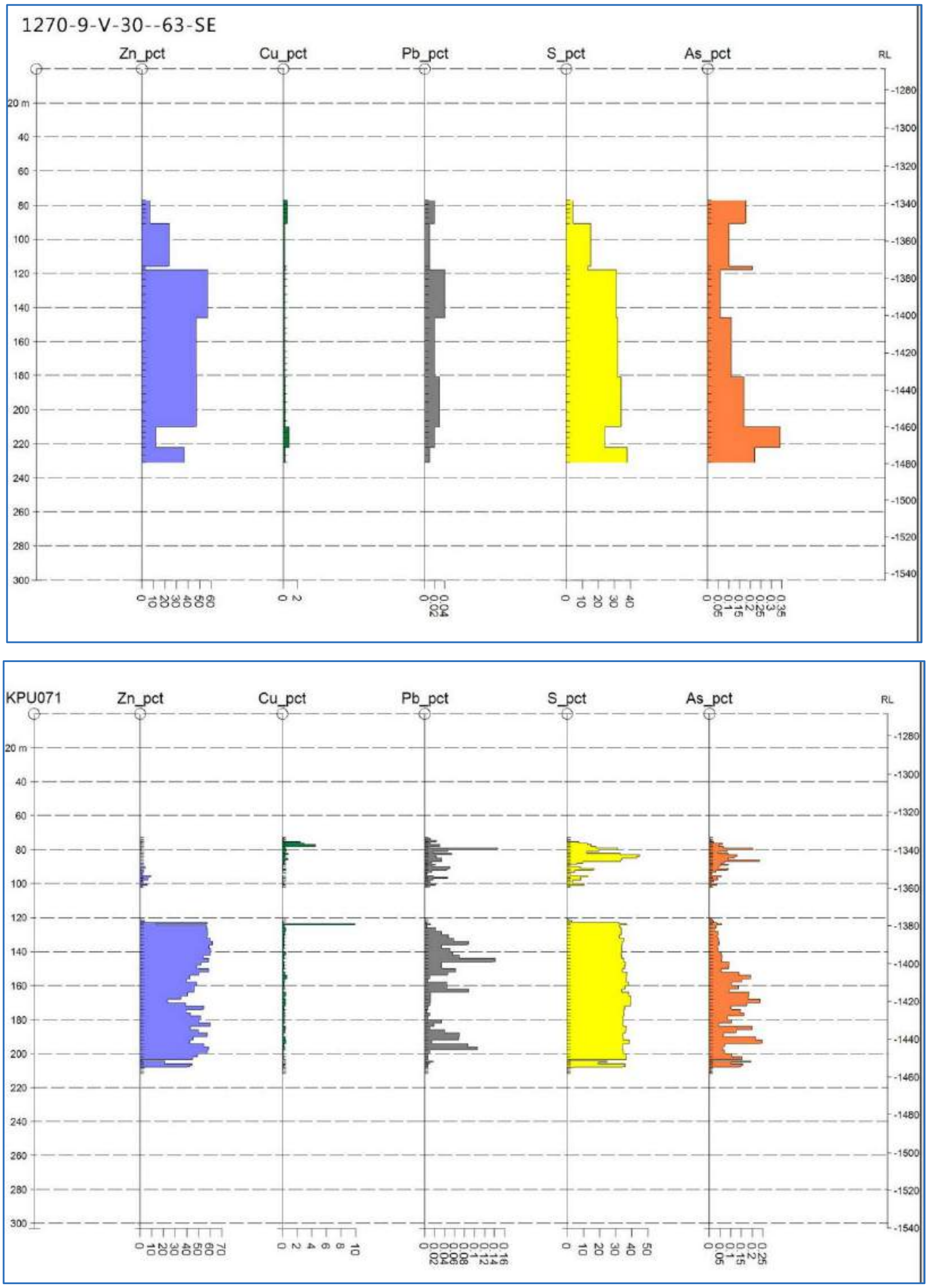


Figure 12.10 Comparison between Gécamines hole 1270/9/V+30/-63/SE and KICO Twin Hole KPU071



12.5 Visual Verification

Mineralization 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 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 QAQC control.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

The Kipushi processing plant originally 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.

Ivanhoe has undertaken two sets of testwork. The first set in 2013 included mineralogy, comminution and flotation testing. The second set in 2015 was to examine Dense Media Separation (DMS). A review of potential process routes was undertaken by Ivanhoe that suggested, given the favourable density differences between massive sulphides and the gangue material, Heavy Media or DMS was considered as a highly likely alternate to flotation, potentially providing lower capital and operating costs.

OreWin undertook a review of the metallurgical testwork carried out by Ivanhoe. This included a review of the testwork procedures and results, and a visit in September 2015 to the Mintek Metallurgical Laboratory (Mintek) in Johannesburg, South Africa by OreWin's Principal Process Consultant.

13.1 Metallurgical Testwork - 2013

In 2013, approximately 60 kg of quarter-core was delivered to the Mintek laboratory for testwork that included mineralogy, comminution and flotation testing.

The composite sample head analysis was 38% Zn, 0.78% Pb, 0.4% Cu, 34% S, and 12% Fe. Mineralogy of the sample showed sphalerite being predominant (65.9%), followed by pyrite (24%), with galena and chalcopyrite present in minor quantities. The major gangue was silica and carbonaceous minerals. The sphalerite and galena are coarse grained, grains up to 1 mm and 0.5 mm respectively. Chalcopyrite is relatively fine grained, less than 0.04 mm.

Comminution testing showed the mineralisation to be soft, with Bond Ball Work Index of 7.8 kWh/t and SAG Milling Comminution (SMC) parameters A x b of 105. Preliminary flotation tests indicated a zinc rougher recovery of 87% at 56% concentrate grade with 50% passing 75 µm grind.

13.1.1 Metallurgical Testwork – 2015 Sample Selection, Preparation and Compositing

A metallurgical sampling and testwork campaign was conducted in early 2015. Testwork was again carried out at the Mintek Laboratory in Johannesburg, South Africa. The Big Zinc zone was the primary focus of this campaign. Six holes intersecting the Big Zinc zone were selected and core intervals were composited for metallurgical and mineralogical investigations. The samples came from hole numbers; KPU001, KPU003, KPU042, KPU051, KPU058, and KPU066. The drill core was selected to represent most mineralisation types in the Big Zinc zone; including but not limited to massive brown sphalerite (MSB), massive sulphide mixed (MSM), dolomite (SDO), etc. The target head grade for the composite sample was 37% Zn, based on the assayed intervals of the drill cores.

Drill core intersections used to make up the composite sample are shown in Table 13.1.

Table 13.1 Composite Sample Testwork Details

HOLE ID	Sample Length	Sample Type	Hg (ppm)	Ag (ppm)	As (ppm)	Cu (%)	Ge (ppm)	Pb (%)	Zn (%)	S (%)	Measured Density	Measured Mass	Dominant Mineral	
Individual Drillholes														
KPU001	12.0	N_CORE	45.69	19.54	591.03	0.12	50.60	0.04	58.5	33.4	4.07	44.6	ZN	
KPU001	9.8	N_CORE	1.50	0.68	263.90	0.01	6.54	–	0.4	1.9	2.89	27.4	SDO	
KPU001 Total	**21.8**	**N_CORE**	**28.85**	**12.36**	**466.40**	**0.08**	**33.81**	**0.03**	**36.4**	**21.4**	**3.64**	**72.0**	–	
KPU003	12.0	N_CORE	52.32	15.65	651.41	0.16	83.69	1.57	53.7	32.3	4.00	43.0	PYR/ZN	
KPU003	7.5	N_CORE	0.16	0.18	479.67	0.02	9.00	–	0.0	1.1	2.89	21.7	SDO	
KPU003 Total	**19.5**	**N_CORE**	**34.82**	**10.46**	**593.81**	**0.12**	**58.64**	**1.04**	**35.7**	**21.9**	**3.65**	**64.7**	–	
KPU042	10.4	H_CORE	54.68	8.92	823.24	0.32	271.81	0.06	52.6	32.8	3.99	40.7	MSM	
KPU042	10.8	H_CORE	3.92	2.01	799.29	0.23	39.62	–	2.4	5.2	3.05	33.6	SDO	
KPU042 Total	**21.2**	**H_CORE**	**31.71**	**5.80**	**812.40**	**0.28**	**166.71**	**0.03**	**29.9**	**20.3**	**3.57**	**74.3**	–	
KPU051	12.0	N_CORE	45.19	9.05	1,396.32	0.24	55.13	0.07	49.9	35.8	4.42	50.3	MSM	
KPU051	6.0	N_CORE	0.03	1.37	832.00	0.07	1.68	–	0.1	0.8	2.94	15.6	SDO	
KPU051 Total	**18.0**	**N_CORE**	**34.48**	**7.23**	**1,262.54**	**0.20**	**42.46**	**0.05**	**38.1**	**27.5**	**4.05**	**65.9**	–	
KPU058	16.5	N_CORE	40.26	24.19	1,034.22	0.96	75.56	0.04	54.2	34.4	4.08	47.9	MBS/MSM	
KPU058	6.0	N_CORE	1.10	5.00	3,477.90	0.33	1.95	0.00	0.1	8.6	3.09	20.8	SDO	
KPU058 Total	**22.5**	**N_CORE**	**28.40**	**18.37**	**1,774.58**	**0.77**	**53.26**	**0.03**	**37.8**	**26.6**	**3.86**	**68.7**	–	
KPU066	14.7	N_CORE	–	–	–	0.13	–	–	47.6	–	4.23	54.6	MBS	
KPU066	2.5	N_CORE	–	–	–	0.00	–	–	1.5	–	2.91	6.4	SDO	
KPU066 Total	**17.2**	**N_CORE**	–	–	–	**0.12**	–	–	**42.8**	–	**4.26**	**61.0**	–	
Composite Sample														
ZN	77.6	–	38.11	12.53	738.68	0.32	82.90	0.27	52.5	27.3	4.14	281.0	–	69%
SDO	42.5	–	1.59	1.72	1034.52	0.13	14.12	0.00	0.9	3.5	2.97	125.6	–	31%
Total	**120.1**	–	**26.83**	**9.19**	**830.06**	**0.26**	**61.66**	**0.19**	**36.6**	**19.9**	**3.84**	**406.5**	–	**100%**

Approximately 407 kg of NQ (45 mm diameter) half core material was selected for the testwork and sent to the laboratory. The core was composited by crushing to –20 mm and then thoroughly blended before riffle-splitting a sub-sample of 220 kg. The 220 kg sub-sample was further split as follows:

- A 10 kg fraction was removed and crushed further to –1.7 mm and further split, prepared and submitted for head chemical analysis and mineralogical investigations.
- 3 x 70 kg batches were then prepared. Two batches were individually crushed to –12 mm and –6 mm respectively, the third batch was reserved already at –20 mm.

The remainder of the master composite sample (approximately 294 kg) was reserved for future testwork.

13.1.2 Head Assay and Mineralogy

Head assays results are presented in Table 13.2.

Table 13.2 Kipushi Composite Sample Head Analysis

	Zn (%)	Pb (%)	Fe (%)	Ca (%)	Si (%)	Cu (%)	Mg (%)	S (%)
Average Assay	40.1	1.45	5.97	6.20	1.73	0.27	3.55	25.5

Mineralogical investigations were conducted on the crushed material, at 100% passing 1.7 mm. The main minerals encountered in order of abundance were sphalerite (67%), galena (2%) and chalcopyrite (1%). The main gangue minerals in the sample were dolomite (18%), pyrite (8%) and quartz (3%).

13.1.3 Dense Media Separation and Shaking Table Testwork

Dense medium separation (DMS) is often used as a simple concentration technique for materials with sufficient density differentials between waste and mineralised material.

DMS washability profiles were evaluated in the laboratory at three feed crush sizes using a combination of heavy liquid separation (HLS) and shaking tables. Fine material (-1 mm), mainly generated during crushing, was screened off ahead of HLS separation and tested on bench scale shaking tables (shaking tables provide a laboratory scale simulation of a commercial spiral plant). Fine material of -1 mm is not suitable for treatment by HLS.

HLS or sink-float analysis is a laboratory scale characterisation method that uses heavy liquid as a medium of separation. The density of the liquid is adjusted by adding a fine powder such as ferro silicon (FeSi). Representative 20 kg sub-samples of the -20+1 mm, -12+1 mm and -6+1 mm fractions were subjected to HLS testwork at density cut points between 2.6 g/cm^3 and 3.8 g/cm^3 at increments of 0.1 g/cm^3. The HLS results indicated that a density cut point of 3.1 g/cm^3 was optimal in all cases. The results are summarised in Table 13.3.

Table 13.3 Summary of HLS Results at a Density Cut Point of 3.1 g/cm3

Size Fraction	Head Grade		Concentrate (Sinks)			Tailings (Floats)	
	Calculated Zn (%)	Calculated Ca (%)	Mass yield (%)	Zn Grade to conc (%)	Zn Rec (%)	Mass yield (%)	Ca rejection (%)
-20+1mm	40.3	5.87	72.6	55.4	99.7	27.4	91.6
-12+1mm	39.6	6.00	74.1	53.2	99.6	25.9	86.0
-6+1mm	40.6	5.93	72.5	55.8	99.6	27.5	90.6

The summary shows that across all three crush sizes, zinc recoveries of over 99% were achieved at a product grade of approximately 55% Zn (based on HLS feed only). Gangue material, mainly dolomite, was rejected to the float stream at an average mass percentage of 26% for all three crush sizes.

Finer crushing does not appear to effect zinc upgrading or gangue rejection, it does however increase fines generation which bypass the HLS and are treated on the less efficient shaking tables.

The fine material (-1 mm), removed ahead of the HLS was tested on a bench scale shaking table (the shaking table is a bench scale technique used to evaluate the commercial application of spirals) to evaluate the separation of gangue from mineralized material. The shaking table results for the fines associated with the three crush sizes are presented in Table 13.4.

Table 13.4 Shaking Table Results Summary

	Head Grade		Concentrate (conc1-tails1)			Tails (slimes+tails2)		
Size Fraction	Calculated Zn (%)	Calculated Ca (%)	Mass yield (%)	Zn grade in concentrate (%)	Zn recovery (%)	Mass yield (%)	Zn grade (%)	Zn rec loss (%)
-20+1mm	43.3	4.99	48.1	54.9	61.0	51.9	32.6	39.0
-12+1mm	42.9	5.08	49.0	54.2	61.9	51.0	32.0	38.1
-6+1mm	42.5	5.41	49.0	55.6	64.0	51.0	30.0	36.0

The shaking table results indicate that a zinc concentrate product with a recovery of approximately 61% at a grade approximately 55% Zn was achieved for the -1 mm fraction at all three crush sizes (based shaking table feed). Recovery losses of between 36% and 39% to spiral tails were mainly due to slimes and the inefficient recovery method (shaking table).

13.1.4 Discussion and Conclusion

Performance across the HLS and the shaking table, as a function of feed, is the same for all three crush sizes. The HLS circuit achieved 99% recovery at a concentrate grade approximately 55% Zn, while the shaking table achieved 61% recovery at a concentrate grade approximately 55% Zn. The difference in overall performance of the three crush sizes is the mass percentage reporting to the -1 mm fines fraction processed through the less efficient shaking tables. The relatively low mass percentage of the -20 mm crush size material reporting to the shaking tables makes this result far superior as only 10% of feed bypass the HLS compared to 22% and 32% of the -12 mm and -6 mm samples respectively.

Testwork concentrate specifications are shown in Table 13.5.

Penalty elements analysis are generally within acceptable limits.

Table 13.5 Concentrate Analysis

Description	Zn (%)	Pb (%)	Fe (%)	Ca (%)	Si (%)	Cu (%)	Mg (%)	S (%)	Au (ppm)	Ag (ppm)	Ge (ppm)	Cd (ppm)	Sb (ppm)	Hg (ppm)	As (ppm)	Cl (ppm)	F (ppm)
Final Concentrate	55.4	2.1	7.5	0.7	1.2	0.3	0.3	29.3	0.3	33.4	82.7	2159	9.8	37.9	874.2	260.3	861.3

A summary of metallurgical testwork composite and testwork results is shown in Table 13.6.

Table 13.6 Summary of Metallurgical Testwork Composite and Testwork Results

		Sample Length (m)	Zn (%)	Measured Density	Sample Mass (kg)	Mass (%)
Total Samples	**Mineralised**	77.63	52.53	4.14	281	69
	Non-Mineralised	42.49	0.85	2.97	126	31
	Average	98.96	36.57	3.84	407	100
Testwork	**Concentrate**	–	55.50	–	–	70
	Tailings	–	6.10	–	–	30
	Head Grade	–	40.60	–	–	100
KPU001	**Mineralised**	12.00	58.55	4.1	43.5	62
	Non-Mineralised	9.76	0.38	2.9	25.1	38
	Average	21.76	36.38	3.6	68.6	100
KPU003	**Mineralised**	12.00	53.73	4.0	42.7	66
	Non-Mineralised	7.50	0.02	2.9	19.3	34
	Average	19.50	35.72	3.7	61.9	100
KPU042	**Mineralised**	10.39	52.56	4.0	36.9	55
	Non-Mineralised	10.77	2.44	3.0	29.0	45
	Average	21.16	29.88	3.6	65.9	100
KPU051	**Mineralised**	12.00	49.87	4.4	47.2	76
	Non-Mineralised	6.00	0.10	2.9	15.6	24
	Average	18.00	38.07	4.1	62.8	100
KPU058	**Mineralised**	16.50	54.19	4.1	59.8	70
	Non-Mineralised	6.01	0.14	3.1	16.4	30
	Average	22.51	37.82	3.9	76.3	100
KPU066	**Mineralised**	14.74	47.64	4.2	57.6	89
	Non-Mineralised	2.45	1.50	2.9	6.3	11
	Average	17.19	42.79	4.3	64.0	100

13.1.5 Recommendations

It is recommended that KICO analyses the testwork and defines a process option that can be applied to further studies and metallurgical investigation. The following testwork is recommended to support future studies.

- DMS testwork on variability samples over a range of zinc feed grades and locations.
- If available, a bulk sample and pilot programme is recommended using DMS and spirals. This is to confirm the design criteria across a DMS / Spiral circuit.
- Mineralogy of feed and detailed concentrate analysis is suggested in order to ensure a suitable geometallurgical model can be created.

14 MINERAL RESOURCE ESTIMATES

On behalf of KICO, The MSA Group (MSA) has completed a Mineral Resource estimate for the Kipushi Zn-Cu Project (Kipushi). Kipushi is located in the town of Kipushi in the Katanga Province in The Democratic Republic of the Congo (DRC). Kipushi is an historical mine currently under care and maintenance that was previously operated by Gécamines.

To the best of the Qualified Person's knowledge there are currently no title, legal, taxation, marketing, permitting, socio-economic or other relevant issues that may materially affect the Mineral Resource described in the Kipushi 2016 PEA, aside from those already mentioned in Section 4.

The Mineral Resource estimate incorporates drilling data collected by KICO from March 2014 until November 2015 inclusive, which, in the Qualified Person's opinion, were collected in accordance with The Canadian Institute of Mining, Metallurgy and Petroleum (CIM) "Exploration Best Practices Guidelines". Previous drilling work completed by 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 3 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 between March 2014 and November 2015, with the cut-off date for data included in this estimate being 16 December 2015. 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, 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 and 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, hand written 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;
  - Drillhole name – this contains information on the section number, bearing and dip of the drillhole,
  - Easting and northing and local mine coordinates
  - Elevation – where elevation was not recorded on the collar sheet, the elevation was gleaned from sections,
  - Section name and level,
  - Start and end date of the drilling,
  - Comments,
  - Core recovery in metres and percentage,
  - Collar inclination and azimuth - the drillhole name itself contains information on the dip and direction at the hole collar that could be used in cases where the collar coordinates were not available elsewhere.
- Downhole surveys;
  - Drillhole name,
  - Depth of survey point,
  - Magnetic bearing,
  - True bearing - the hard copy data exists as bearings relative to north or south and so the azimuth was calculated in degrees and added to the database,
  - Dip,
  - Comments.

- Where there are no survey data for a drillhole, the collar survey inclination and bearing were used as the downhole survey.
- Assays;
    - Drillhole name,
    - Start and end depth of the sample (from, to),
    - Grades of Cu, Pb, Zn, S, Fe, As,
    - Units of assays,
    - Density,
    - Comments.
- Lithological log;
    - Drillhole name,
    - Start and end depth of the record (from, to),
    - Two tiers of lithology were captured as Lith1 and Lith2 fields based on the free form geological descriptions in the log,
    - Colour,
    - Comments.
- Mineralization log;
    - Drillhole name
    - Start and end depth of the record (from, to),
    - Four levels of mineralization relating to the most abundant (Min1_code) to the least abundant (Min4_Code).

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 mineralization. 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. 33 of the drillholes 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 between March 2014 and November 2015. 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 mineralized envelope. The original Gécamines data was replaced with the KICO re-sampled data for 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 more completely sampled in the lower grade mineralization than the Gécamines holes and thus any short range discontinuities in the lower grade mineralization 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 & 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 Kipushi Sample Database

Element	Element Symbol	Units	Lower Detection Limit
Gold	Au	ppb	1
Platinum	Pt	ppb	20/50
Palladium	Pd	ppb	20/50
Mercury	Hg	ppm	0.01/10
Silver	Ag	ppm	5 or 0.05
Arsenic	As	ppm	10
Cadmium	Cd	ppm	10
Cobalt	Co	ppm	10
Copper	Cu	ppm	50
Germanium	Ge	ppm	5
Lead	Pb	ppm	20
Zinc	Zn	ppm	50
Rhenium	Re	ppm	0.1
Sulphur	S	%	0.01
Nickel	Ni	ppm	20/50
Molybdenum	Mo	ppm	5
Uranium	U	ppm	0.5
Vanadium	V	ppm	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 that 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. -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 Mineral Resource Qualified Person. The validation process consisted of:

- Examining the sample assay, collar survey, downhole survey and geology data to ensure that the data are complete for all of the drillholes.
- Examination of the assay and density data in order 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 mineralization.
- 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 mineralization worth sampling and thus the core interval is barren. The Gécamines cores were selectively sampled and samples were only taken when mineralization 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 mineralization worth sampling and thus the core interval is barren. The KICO cores were mostly sampled throughout the length within the mineralized zones and the assignation of zero grades to un-sampled intervals will not result in any biases. For KPU075, a large part of the mineralized 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 mineralized zone (as observed by the mineralization log).
- 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 also set to null and the copper and/or zinc values were retained.
- Several of the KICO specific gravity measurements are outside of expected limits. Two measurements are less than 2.1 g/cm^3 and were set to a null value ("-") by MSA. Two measurements are greater than 5.25 g/cm^3 (5.77 and 6.98 g/cm^3) 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 mineralization.
- 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 mineralized 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 mineralized zones. These drillholes are 1132/18/V+6/-60/SE, which does not fit the mineralized zones, 1138/1/R+31/-70/SW which plots well within the Fault Zone footwall, 1138/1/R+31/-70/NW mineralized intercept plots well within the Série Récurrenté 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 mineralization 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 mineralization 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 mineralization either in dip or strike direction in the Série Récurrenté zone. These holes were not used for grade estimation but were used for defining the extents of the mineralization.
- Eleven Gécamines holes had been twin-drilled and were removed in favour of the KICO drillholes.

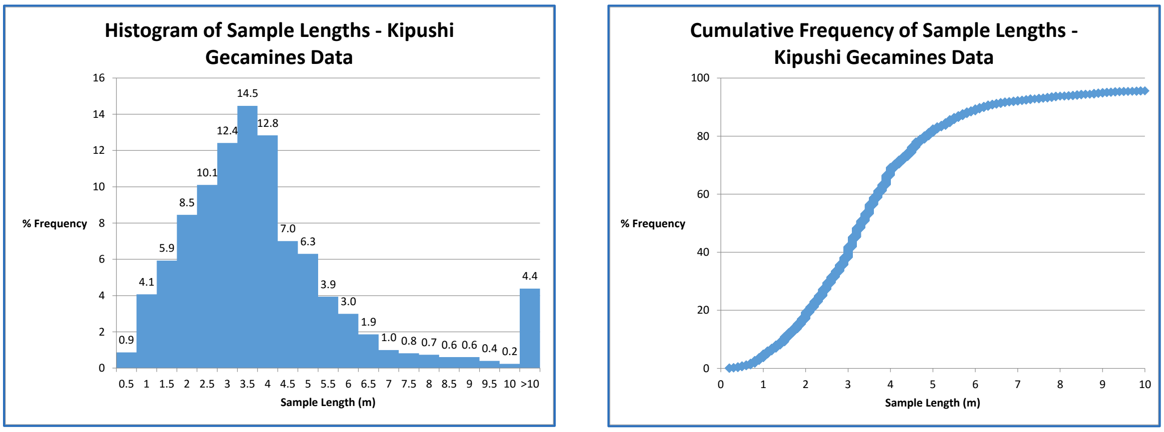
In total there are 93 KICO drillholes that have sampling data. 107 Gécamines drillholes were deemed acceptable for use in the grade interpolation process and an additional 145 Gécamines drillholes were included for the purpose of defining mineralization limits.

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, or Inferred.

14.2.2 Statistics of the Sample Data

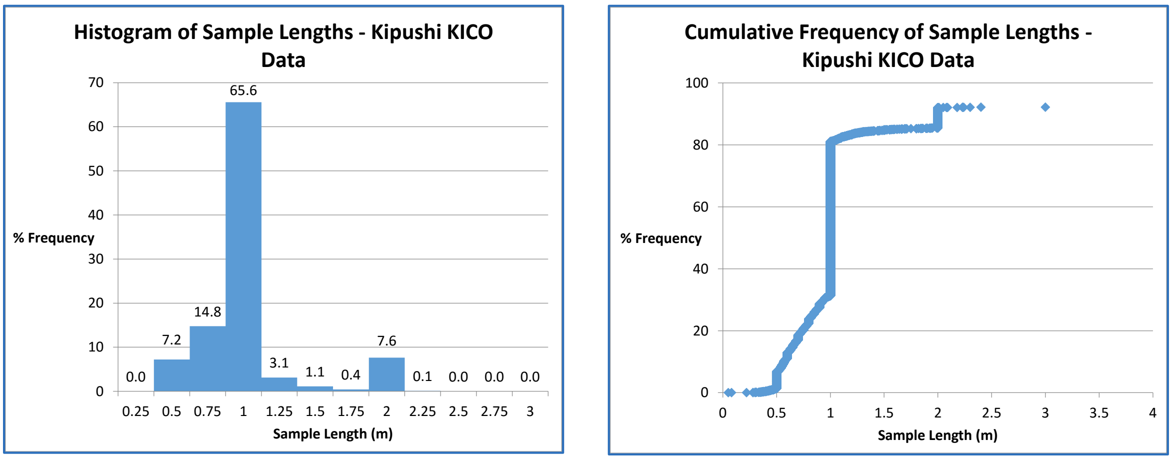
The Gécamines sample data were captured from scans of hard copy hand written and digital logs. Gécamines tended to use a variety of sample lengths considerable longer than what would normally be used in modern practice. In addition, as the database contains composite sample lengths, a number of extreme 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 3 m and 4 m and 82.5% of the sample records are less than 5 m long. As mentioned in Section 14.2.1, Gécamines drillholes that contained well mineralized sample lengths that were excessive were flagged in the estimation database. These holes were used in the construction of the grade shell to define the mineralization 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



The KICO sampling honoured the intensity of mineralization 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



14.2.3 Statistics of the Assay Data

Platinum and palladium assays are of negligible grade, 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.

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. KPU001 and KPU002 were not assayed for uranium.

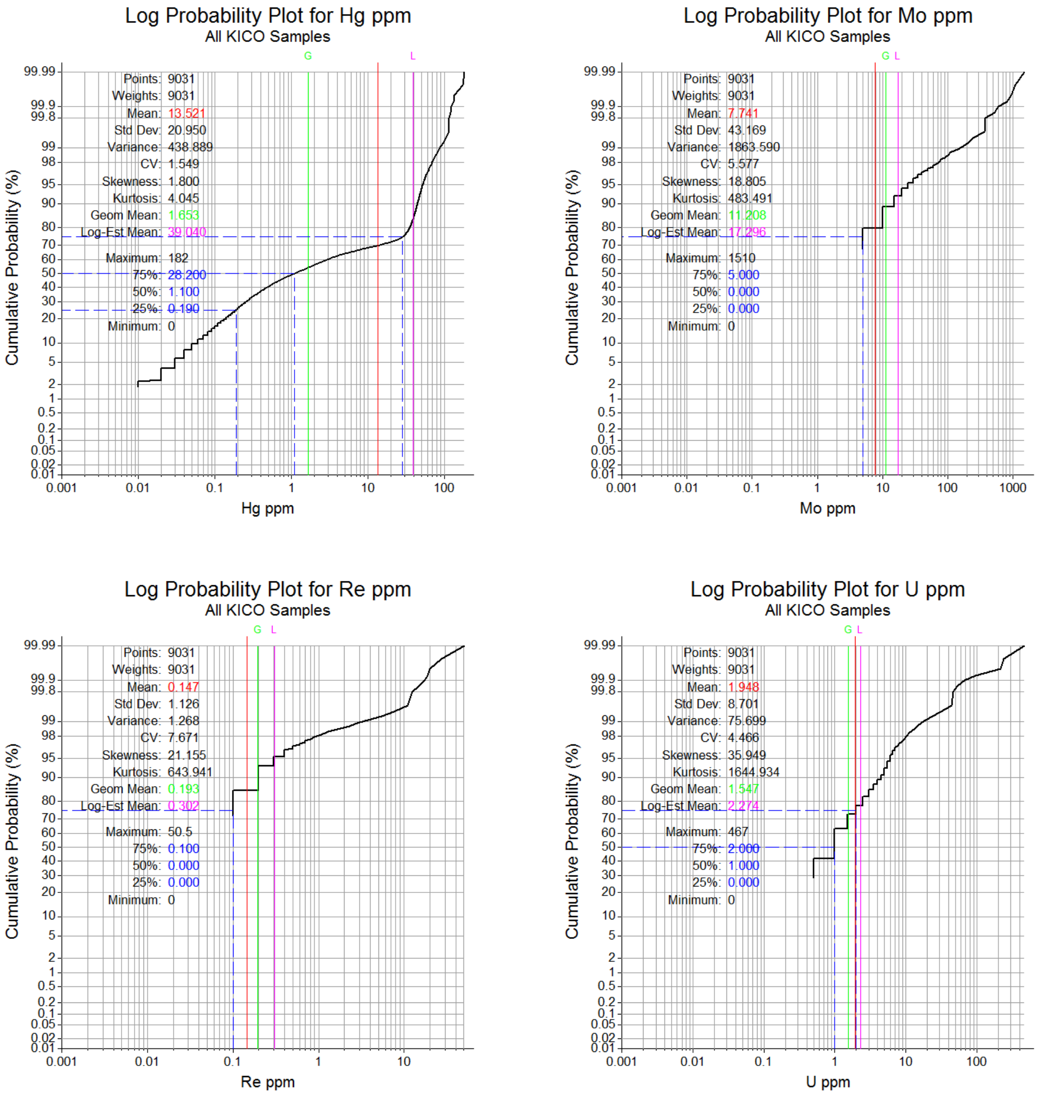
The highest nickel assay is 200 ppm with the majority of the values being below the lower detection limit. Most of the vanadium values are below or slightly above the lower detection limit with the maximum assay being 640 ppm.

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 showed significant grades for these elements, but overall they are low (Figure 14.3). 94% of the mercury assays are less than 50 ppm, 0.5% of the values are above 100 ppm and the highest assay is 182 ppm. 67% of the molybdenum assays are below the lower detection limit (5 ppm), 2.5% are above 50 ppm and the highest assay is 1,510 ppm. 72% of the rhenium assays are below the lower detection limit of 0.10 g/t, 2% are above 1 ppm and the highest assay is 50.5 ppm. Uranium values are generally low with approximately 98% of the values being below 10 ppm and the maximum assay being 467 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.

Further details on the mercury, molybdenum, rhenium, and uranium data are found in Figure 14.3.

Figure 14.3 Log Probability Plot for Mercury, Molybdenum, Rhenium, and Uranium Data



Copper, lead zinc, sulphur, arsenic silver, germanium, cobalt, cadmium and density were considered of importance to the Kipushi Zn-Cu Project and these were examined in greater detail and estimated into the Mineral Resource block model. Iron was not considered.

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%	2,181	2.42	0.01	60.80
Pb%	1,917	0.68	0.01	16.40
Zn%	2,154	10.05	0.01	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	–	–	–

*[1] Where re-sampled Gécamines assays have been replaced with KICO assays

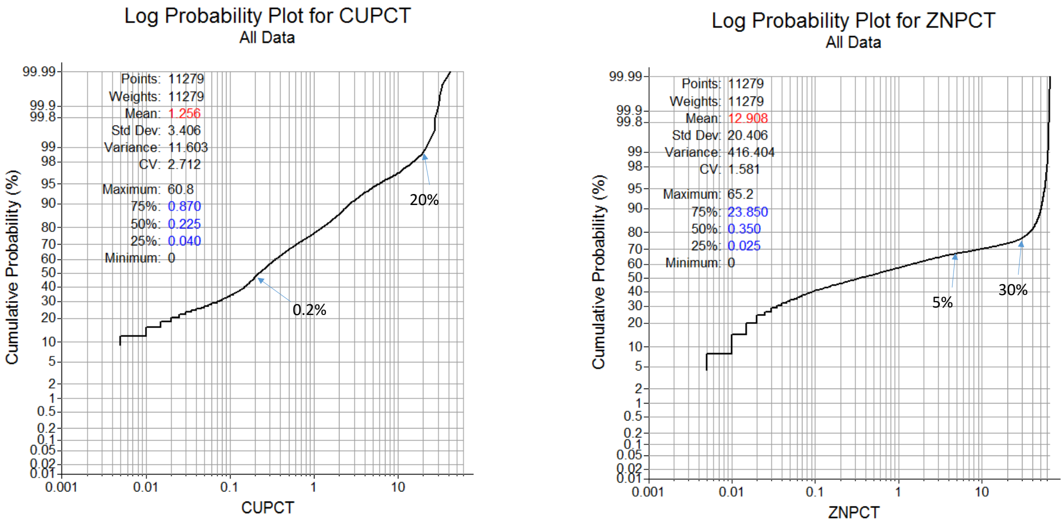
Table 14.4 Summary of the Raw Validated Sample Data for the KICO Drillholes

Variable	Number of Assays	Mean Value	Minimum Value	Maximum Value
Cu%	9,031	0.99	0.00	33.30
Pb%	9,031	0.17	0.00	17.90
Zn%	9,031	13.72	0.00	65.20
S%	9,031	13.15	0.00	51.70
As%	9,031	0.19	0.00	14.70
Ag g/t	9,031	12.9	0.00	3,260.0
Ge g/t	9,031	25.8	0.0	755
Co ppm	9,031	49	0.0	25,300
Cd ppm	9,031	702	0	7,850
Density g/cm^3	5,203	3.38	2.13	5.21

The Gécamines database does not contain values for silver, germanium, copper or cadmium as well as some of the copper, lead, zinc, sulphur, 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 mineralized zones, unlike the Gécamines sampling that was selective aimed at higher copper grade mineralization.

Several zones of mineralization have been identified by Gécamines and KICO. The zones of mineralization 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. Approximately 23% of the combined valid Gécamines and KICO samples are above 20% zinc and only 1% of the samples are greater than 20% copper (Figure 14.4).

Figure 14.4 Log Probability Plot for Copper and Cumulative Distribution for Zinc Sample Assays



14.2.3.2 Bivariate Analysis

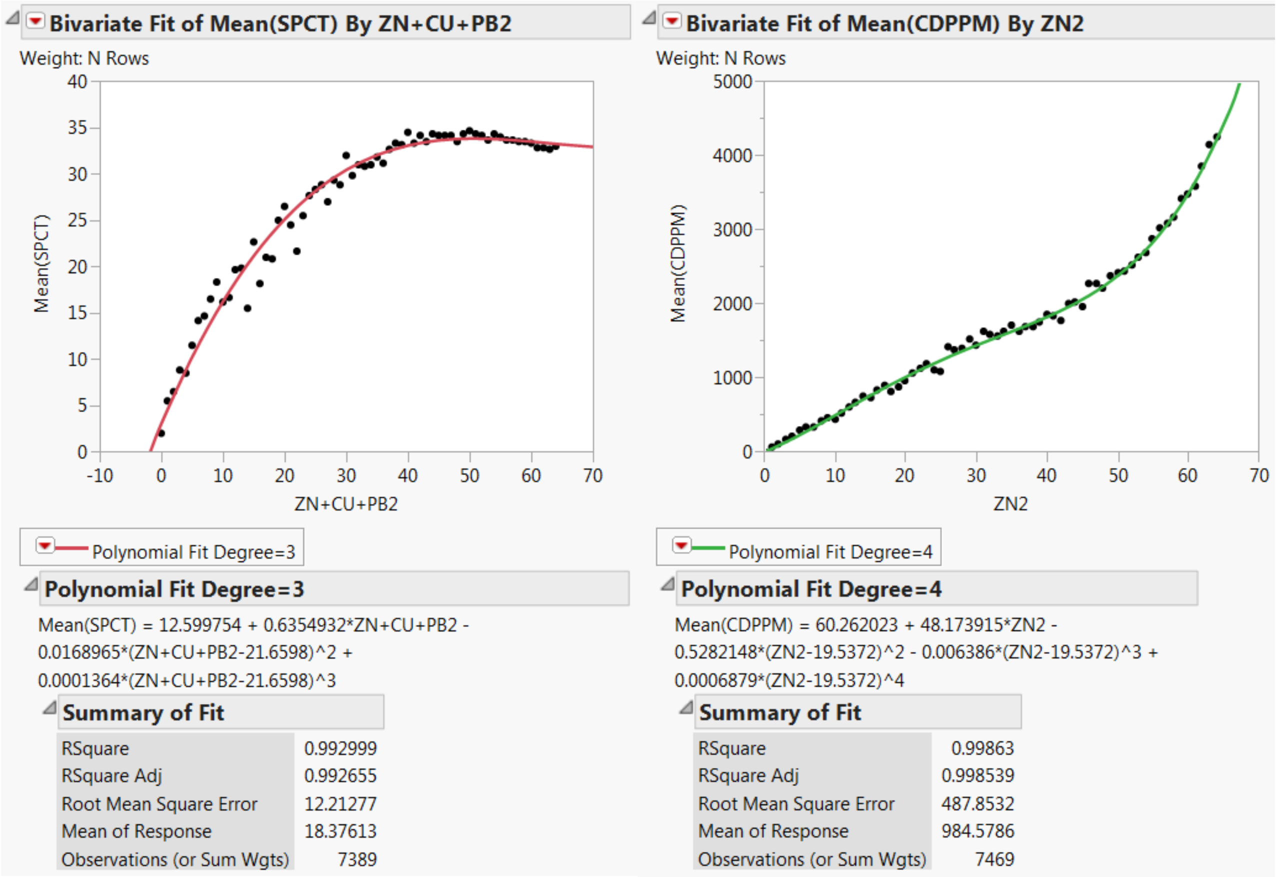
Scatterplots were made that compare the grades of individual elements against one another. The scatterplots for the total data show various relationships that indicate mixed mineralization domains. Several mineralization styles at Kipushi exist, the zinc-rich zones resulting in different bivariate relationships than the copper-rich zones. No clear relationships were found between copper, lead, zinc, and cobalt. Mixed linear relationships are evident between copper and sulphur, zinc and sulphur, 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 sulphur 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 sulphur and between zinc and cadmium. Sulphur 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 14.5). A third order polynomial line was fitted to the sulphur 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 14.5 Sulphur and Cadmium Regressions

Sulphur vs Cu-Zn-Pb **Cadmium vs Zn-Pb**



Bivariate Fit of Mean(SPCT) By ZN+CU+PB2

Weight: N Rows

Polynomial Fit Degree=3

Mean(SPCT) = 12.599754 + 0.6354932*ZN+CU+PB2 - 0.0168965*(ZN+CU+PB2-21.6598)^2 + 0.0001364*(ZN+CU+PB2-21.6598)^3

Summary of Fit

RSquare	0.992999
RSquare Adj	0.992655
Root Mean Square Error	12.21277
Mean of Response	18.37613
Observations (or Sum Wgts)	7389

Bivariate Fit of Mean(CDPPM) By ZN2

Weight: N Rows

Polynomial Fit Degree=4

Mean(CDPPM) = 60.262023 + 48.173915*ZN2 - 0.5282148*(ZN2-19.5372)^2 - 0.006386*(ZN2-19.5372)^3 + 0.0006879*(ZN2-19.5372)^4

Summary of Fit

RSquare	0.99863
RSquare Adj	0.998539
Root Mean Square Error	487.8532
Mean of Response	984.5786
Observations (or Sum Wgts)	7469

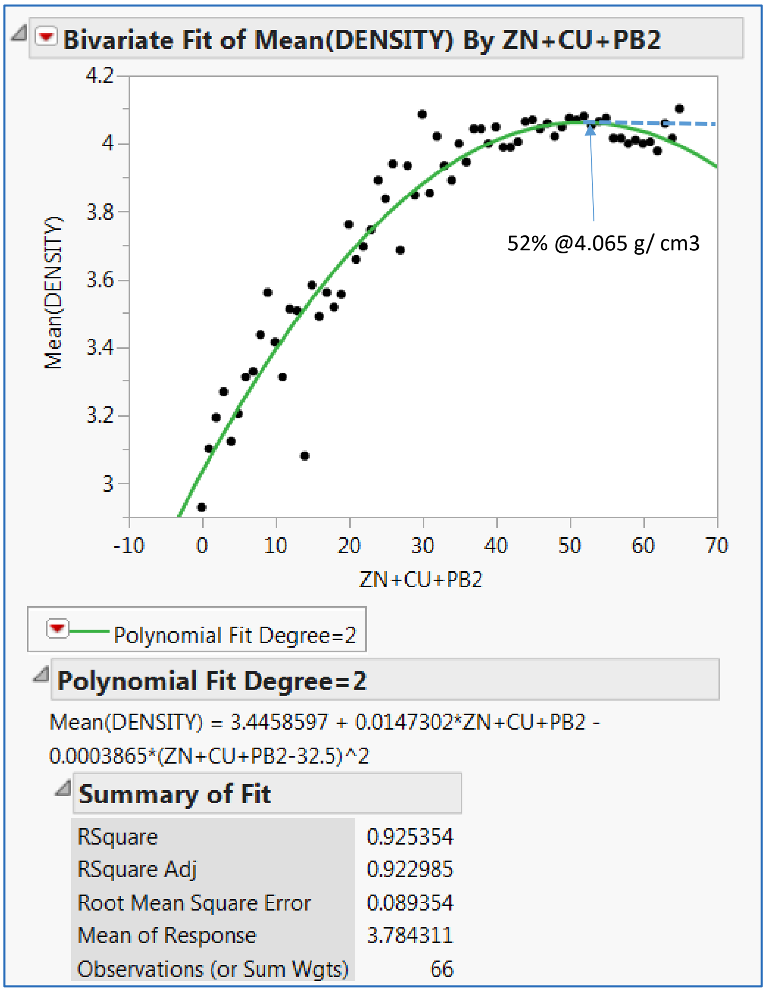
Density Determination

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 of 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 of 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 mineralization at Kipushi, however the mineralization 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 14.6. The regression is capped at 52% Cu+Zn+Pb and a constant of 4.065 g/cm^3 was applied to samples above this 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 14.6 Density Regression



Bivariate Fit of Mean(DENSITY) By ZN+CU+PB2

Polynomial Fit Degree=2

Mean(DENSITY) = 3.4458597 + 0.0147302*ZN+CU+PB2 - 0.0003865*(ZN+CU+PB2-32.5)^2

Summary of Fit

RSquare	0.925354
RSquare Adj	0.922985
Root Mean Square Error	0.089354
Mean of Response	3.784311
Observations (or Sum Wgts)	66

14.2.4 Summary of the Exploratory Analysis of the Raw Dataset

- KICO assays below detection limit were assigned zero values, they existing 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 mineralized 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 mineralized zone as defined by the mineralization log.

- 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 93 KICO drillholes that have sampling data. 107 Gécamines drillholes were deemed acceptable for use in the grade interpolation process and an additional 145 Gécamines drillholes were included for the purpose of defining mineralization 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 or Inferred.
- Copper, lead zinc, sulphur, arsenic silver, germanium, cobalt, cadmium, and density are considered of importance to the Kipushi Zn-Cu 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 sulphur 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 mineralization 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 Mineralized Zones

The mineralization at Kipushi comprises sulphide replacement bodies within the Kakontwe Sub-Group dolomites and Série Récurrenté Sub-Group dolomitic shales of the Nguba Group.

Two zones of zinc-rich mineralization occur, the Big Zinc zone and the Southern Zinc zone, which lie adjacent to the copper-rich Fault Zone mineralization. In places, the Big Zinc mineralization is juxtaposed against the Fault Zone, although in many areas zones barren of significant mineralization occur between them. The Southern Zinc zone is an elongate lense of sphalerite rich mineralization parallel and juxtaposed against the Fault Zone mineralization. A zone of high grade copper, silver and germanium occurs within the Big Zinc zone.

The Fault Zone strikes north–north–east to south–south–west and dips at approximately 70° to the west, with the zinc mineralization forming irregular steeply dipping bodies in the immediate footwall to the Fault Zone. A second zone of copper-rich mineralization occurs in the Série Récurrenté zone which strikes from east to west, is sub-vertical and plunges steeply to the west. Where the Fault Zone and Série Récurrenté zone meet, mineralization tends to be enhanced in a sub-zone known as the Copper Nord Riche zone. A sub-vertical copper-zinc-germanium rich sulphide 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 mineralization, a characteristic of the replacement nature of the mineralization is that it cuts across the layering in places and is not stratabound. For this reason, the mineralization was modelled on the basis of grade thresholds while taking cognisance of the controlling lithological and structural trends.

In total seven zones were modelled as separate wireframes:

- Fault Zone – Zone 1.
- Big Zinc – Zone 2.
- Southern Zinc – Zone 3.
- Série Récurrenté – Zone 4.
- Massive sulphide lense within the Série Récurrenté – Zone 5.
- High grade zone within the Big Zinc – Zone 6.
- Splay Zone - the high zinc-copper-germanium splay from the Fault Zone – Zone 7.

Mineralized 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 mineralization by snapping to the drillhole intercepts. The sections were examined along strike to ensure that the thickness trends of the mineralization were continued from one section to the next. The interpreted strings were then linked to form wireframe solids.

All of the available validated data were used for the construction of the mineralized 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 zone that plunges steeply to the south–west. An isometric view of the wireframe models is shown in Figure 14.7.

Figure 14.7 Isometric View of Kipushi Wireframes and Drillholes (view is approximately to the north–west)

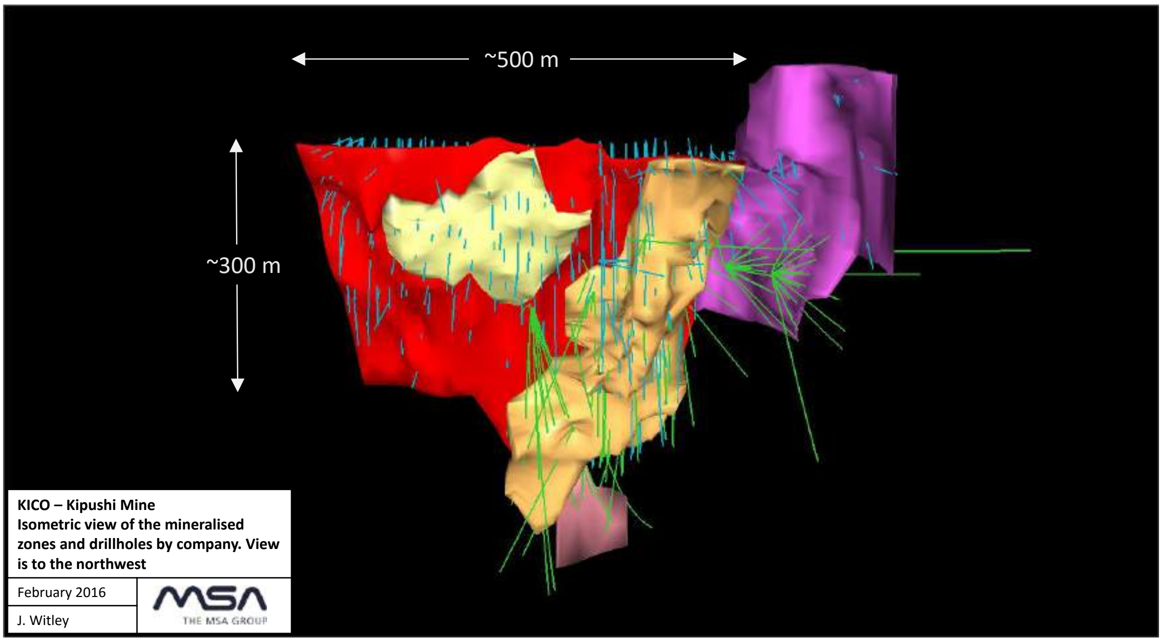


Figure by MSA, 2016.

Red Wireframe = Fault Zone (Zone 1)
Orange Wireframe = Big Zinc (Zone 2)
Beige Wireframe = Southern Zinc (Zone 3)
Violet Wireframe = Série Récurrenté (Zone 4)
Pink Wireframe = Splay 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 50 mX, 50 mY and 50 mZ by weighting by the number of data in each cell and summary statistics were compiled for each mineralized 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	Min	Max	Mean	CV	Skewness
Zone 1						
Cu %	719	0.00	42.25	2.89	1.35	3.0
Pb %	708	0.00	3.72	0.11	3.72	6.7
Zn %	719	0.00	45.55	3.60	1.77	3.0
S %	719	0.00	50.01	11.56	0.87	1.2
As %	533	0.00	9.33	0.24	2.36	7.3
Ag g/t	263	0.00	145.6	18.8	1.30	2.6
Ge g/t	263	0.00	112.7	14.2	1.26	2.2
Co ppm	263	0.00	13,560	193	4.95	9.9
Cd ppm	719	0.00	4839	192	1.90	4.9
Density	719	2.70	4.54	3.24	0.08	1.2
Zone 2						
Cu %	3,450	0.00	60.80	1.09	3.27	7.5
Pb %	3,422	0.00	16.71	0.79	2.92	3.8
Zn %	3,450	0.00	63.60	28.17	0.75	-0.1
S %	3,450	0.00	45.72	23.15	0.59	-0.6
As %	3,410	0.00	5.77	0.18	2.31	7.2
Ag g/t	2,473	0.00	1,031.7	13.7	1.77	13.3
Ge g/t	2,473	0.00	638.4	47.9	1.04	3.2
Co ppm	2,473	0.00	4315	16	6.29	31.2
Cd ppm	3,450	0.00	5,777	1,318	0.84	0.5
Density	3,450	2.46	4.75	3.69	0.12	-0.5

Variable	Number of composites	Min	Max	Mean	CV	Skewness
Zone 3						
Cu %	118	0.00	13.53	1.85	1.12	2.6
Pb %	118	0.00	10.32	1.35	1.55	2.2
Zn %	118	0.00	51.90	17.37	0.87	0.3
S %	118	0.00	39.56	21.35	0.57	-0.3
As %	30	0.00	0.90	0.23	1.31	1.1
Ag g/t	0	–	–	–	–	–
Ge g/t	0	–	–	–	–	–
Co ppm	0	–	–	–	–	–
Cd ppm	118	0.00	2,545	831	0.86	0.2
Density	118	3.04	4.07	3.58	0.10	-0.1
Zone 4						
Cu %	1,234	0.00	26.75	1.93	1.41	3.8
Pb %	1,200	0.00	1.94	0.04	4.72	8.9
Zn %	1,234	0.00	55.00	0.92	3.76	8.0
S %	1,234	0.00	35.61	2.89	1.64	3.8
As %	1,232	0.00	1.70	0.07	2.32	6.4
Ag g/t	341	0.00	57.6	8.0	1.05	2.8
Ge g/t	341	0.00	23.3	0.8	2.63	4.8
Co ppm	341	0.00	1,032	29	2.43	9.4
Cd ppm	1,234	0.00	976	43	3.05	5.0
Density	1,234	2.73	4.06	3.13	0.05	3.2
Zone 5						
Cu %	44	0.87	30.89	12.99	0.70	0.4
Pb %	44	0.00	5.46	0.22	4.10	4.8
Zn %	44	0.02	53.00	14.59	1.23	0.8
S %	44	1.32	31.64	21.65	0.34	-0.8
As %	44	0.01	5.36	0.51	2.10	3.9
Ag g/t	44	5.35	432.3	58.3	1.15	3.8
Ge g/t	44	0.00	67.7	20.7	0.85	1.1
Co ppm	44	0.00	5,058	179	3.79	7.0
Cd ppm	44	0.00	4,308	923	1.30	1.1
Density	44	3.10	4.06	3.71	0.08	-0.4

Variable	Number of composites	Min	Max	Mean	CV	Skewness
Zone 6						
Cu %	177	0.01	31.05	6.48	0.96	1.1
Pb %	177	0.00	13.10	0.77	2.18	3.9
Zn %	177	0.01	54.90	25.94	0.69	0.0
S %	177	0.26	43.25	25.81	0.44	-1.0
As %	177	0.00	0.92	0.20	0.85	1.8
Ag g/t	135	0.00	2,154.9	122.2	2.60	4.6
Ge g/t	135	0.00	339.4	61.4	0.82	2.1
Co ppm	135	0.00	3,880	163	3.19	5.9
Cd ppm	177	0.00	3,690	1,479	0.72	0.2
Density	177	2.67	4.25	3.80	0.11	-1.2
Zone 7						
Cu %	97	0.00	20.16	2.99	1.35	1.8
Pb %	97	0.00	0.05	0.00	2.20	2.9
Zn %	97	0.00	64.27	22.42	1.18	0.5
S %	97	0.00	38.83	24.17	0.53	-1.0
As %	97	0.00	12.43	2.33	1.40	1.3
Ag g/t	97	0.00	82.3	14.0	1.10	1.8
Ge g/t	97	0.00	599.8	125.3	1.32	1.1
Co ppm	97	0.00	2,211	99	2.23	6.9
Cd ppm	97	0.00	5,499	1,480	1.21	0.6
Density	97	2.87	4.63	3.71	0.13	-0.3

For each element in each domain there are a significant number of composites with zero grade. These largely represent un-sampled intervals within the mineralization 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 Zinc zone, this zone being informed only by Gécamines data.

The copper distributions are generally characterised by moderate coefficient of variation (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. has a flat shape). Zinc distributions in the other zones are variable, with high CV's in the copper rich zones, but low to moderate in the high grade more massive copper-rich sulphide 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 low grade population. Sulphur generally has low to moderate CVs, is negatively skewed in the massive sulphide zones (Zones 2, 3, 5, and 6) and positively skewed in the relatively lower sulphur 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 0.07% and 0.20% except for Zone 5 where the mean arsenic grade is 0.51% as a result of several high grade values which have a large impact, there being only 44 composites in this zone. The arsenic grades in the Splay zone (Zone 7) are also high (average of 1.44%).

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 5 and 6). Germanium CVs are low and distributions are moderately positively skewed except for Zone 4 that is generally of low germanium grade with a few values significantly higher than the mean value. Mean germanium values are high in the Big Zinc zone and the massive chalcopyrite and bornite rich zone (Zone 6) within the Big Zinc zone. Very high germanium values occur in the Splay zone (Zone 7).

Cobalt distributions are positively skewed with high CVs caused by a small number of high values.

Density distributions are slightly negatively skewed in the massive sulphide zones and slightly positively skewed in the lower grade copper-rich zones. CVs are low though and the skewness is not severe.

The generally 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 Cutting and Capping

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 made up a disproportional amount of the domain mean. 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).

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 Capped	Mean	CV
Zone 1							
Cu %	719	2.89	1.35	24.34	1	2.86	1.31
Pb g/t	708	0.11	3.72	1.89	9	0.09	3.05
As %	533	0.24	2.36	3.66	8	0.23	2.00
Ge g/t	263	14.2	1.26	69	1	13.8	1.17
Co ppm	263	193	4.95	1,927	10	119	2.44
Cd ppm	719	192	1.90	1,816	1	187	1.70
Zone 2							
Cu %	3,450	1.09	3.27	26.3	3	1.06	2.97
Ag g/t	2,473	13.7	1.77	173	3	13.5	1.49
Ge g/t	2,473	47.9	1.04	340	7	47.5	0.99
Co ppm	2,473	16	6.29	418	8	13	2.99
Zone 3							
Cu %	118	1.85	1.12	8.3	1	1.82	1.05
Zone 4							
Cu %	1,234	1.93	1.41	17.2	8	1.91	1.36
Pb g/t	1,200	0.04	4.72	1.02	7	0.03	3.89
Zn %	1,234	0.92	3.76	19.5	8	0.84	3.18
As %	1,232	0.07	2.32	0.74	14	0.06	1.84
Ge g/t	341	0.8	2.63	9.0	2	0.8	2.33
Co ppm	341	29	2.43	159	6	25	1.25
Cd ppm	1,234	43	3.05	976	8	43	3.05
Zone 5							
Pb g/t	44	0.22	4.1	0.65	4	0.05	3.18
As %	44	0.51	2.10	1.97	1	0.37	1.36
Ag g/t	44	58.3	1.15	266	1	54.5	0.90
Co ppm	44	179	3.79	552	2	98	1.48
Zone 6							
Co ppm	135	163	3.19	714	5	104	1.88
Zone 7							
Co ppm	97	99	2.23	721	2	88	1.56

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. All 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 mineralization 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 or two 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, a plunge of 52° to the south–west within the plane of mineralization was modelled. A plunge of 50° to the west was modelled for the Série Récurrenté zone grade continuity. A vertical plunge was modelled for the Big Zinc zone grade continuity. Although the limits of this zone plunge steeply to the south–west this trend was not evident in the grade continuity analysis. The directions of continuity were kept the same for each attribute within their respective zones.

There were insufficient data to calculate robust variograms for the Southern Zinc zone (Zone 3), the copper rich zone within the Série Récurrenté zone (Zone 5), the copper rich zone within the Big Zinc zone (Zone 6) and the Splay zone (Zone 7). The variograms for the Big Zinc zone were applied to the Southern Zinc zone while adjusting the direction of continuity to the strike of this zone. The variograms for the Fault Zone were applied to Zones 6 and 7 and the variograms for the Série Récurrenté zone were applied to Zone 5.

For the zones that were modelled, the variogram models are robust, there being a number of experimental points at the chosen lag informing the model within the range of the variogram.

For all zones, the variogram ranges are in excess of the general drillhole spacing, with the drillhole spacing being closer than the range of the first variogram structure for most attributes.

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 14.8, Figure 14.9, and Figure 14.10 for Zone 1, Zone 2, and Zone 4 respectively.

Figure 14.8 Zone 1 Copper Variograms

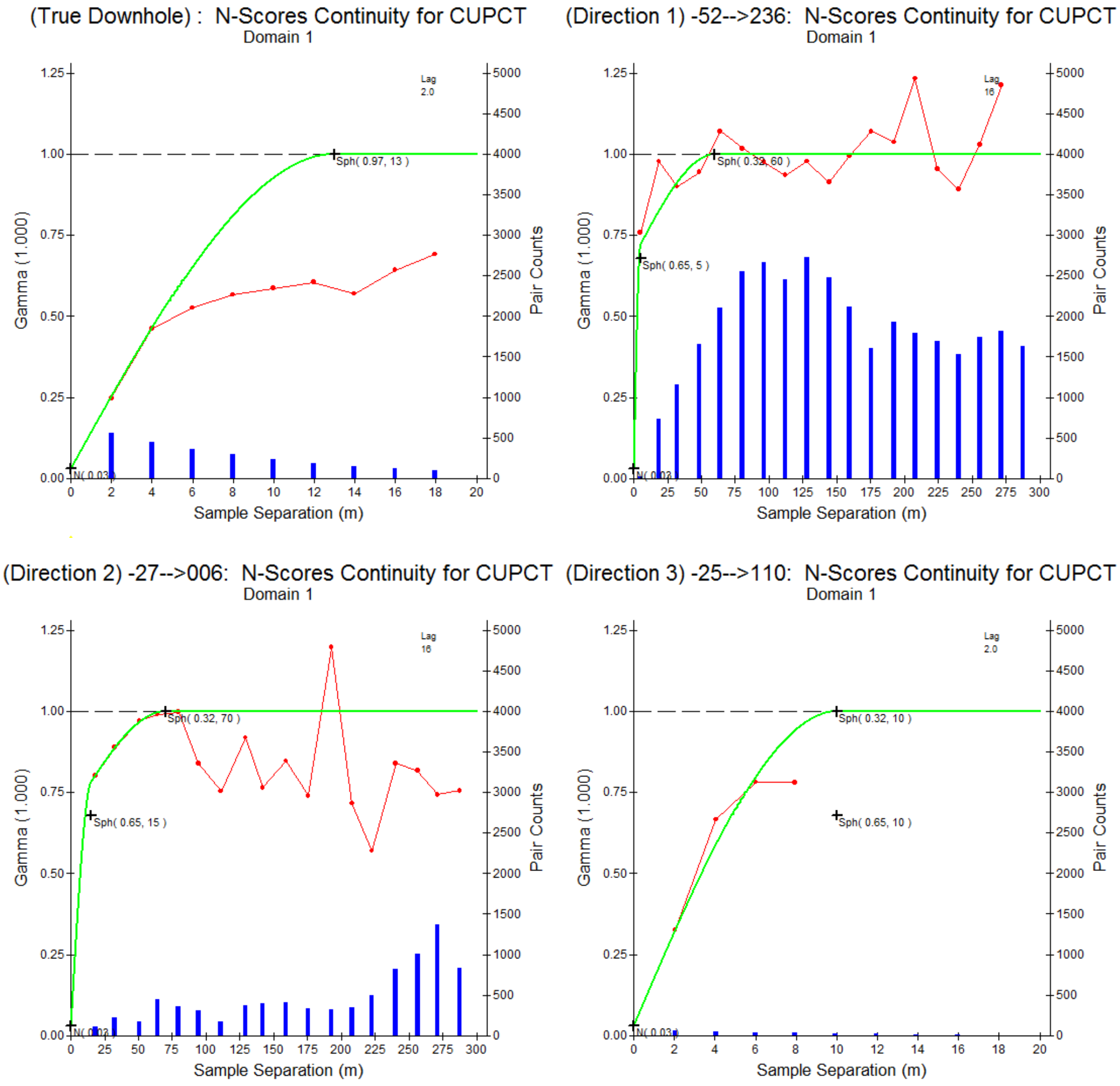


Figure 14.9 Zone 2 Zinc Variograms

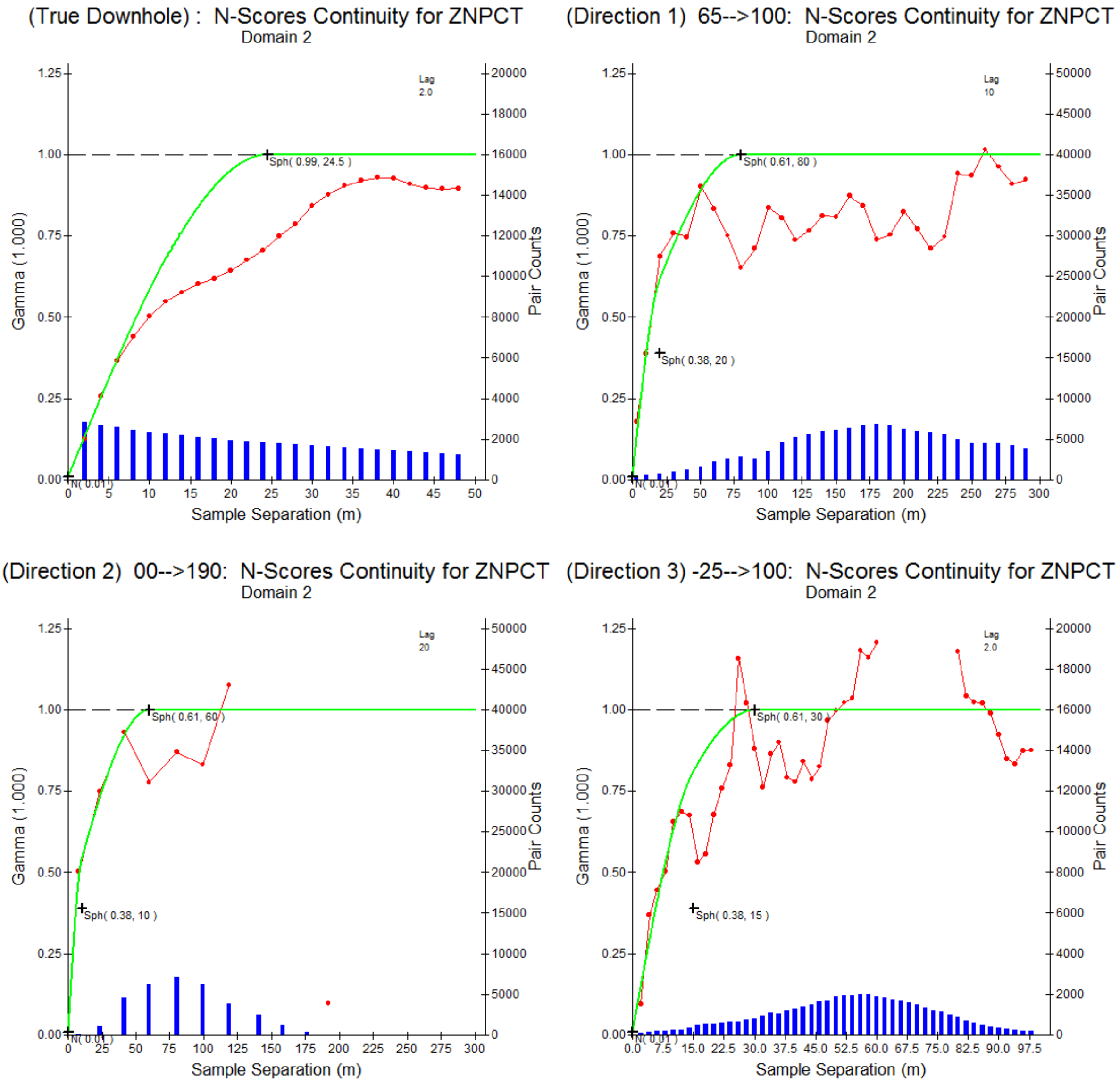


Figure 14.10 Zone 4 Copper Variograms

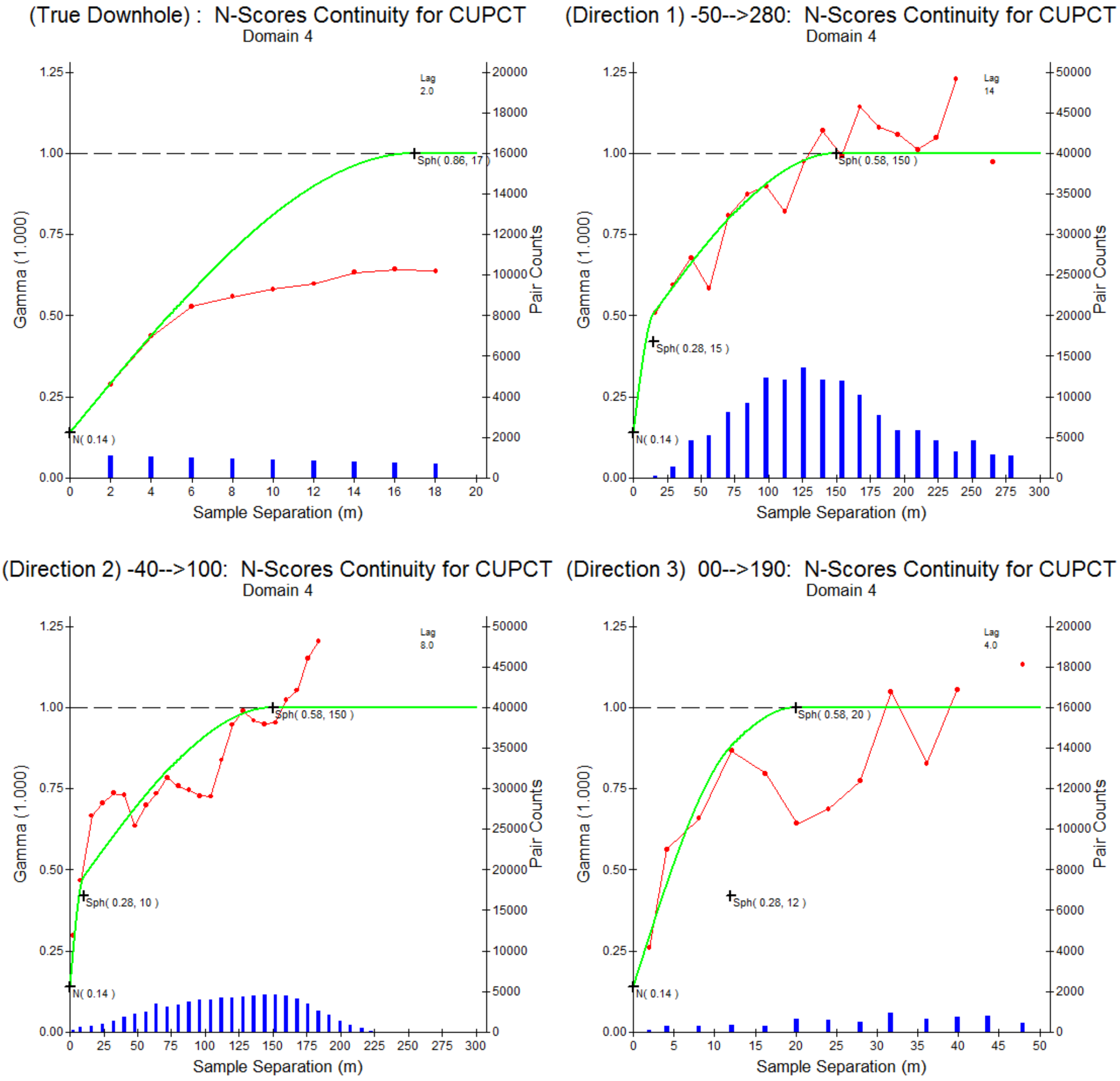


Table 14.7 Variogram Parameters – Kipushi

Attribute	Transform	Rotation Angle			Rotation Axis			Nugget Effect (C^0)	Range of First Structure (R^1)			Sill 1 (C^1)	Range of Second Structure (R^2)			Sill 2 (C^2)
		1	2	3	1	2	3		1	2	3		1	2	3	
Fault Zone																
Cu %	NS	110	115	-60	Z	X	Z	0.04	5	15	10	0.71	60	70	10	0.25
Pb %	NS	110	115	-60	Z	X	Z	0.09	115	115	14	0.91	–	–	–	–
Zn %	NS	110	115	-60	Z	X	Z	0.02	15	50	14	0.55	80	55	14	0.43
S %	NS	110	115	-60	Z	X	Z	0.02	10	25	10	0.54	65	35	10	0.44
As %	NS	110	115	-60	Z	X	Z	0.02	25	25	8	0.98	–	–	–	–
Ag g/t	NS	110	115	-60	Z	X	Z	0.13	25	125	10	0.52	125	125	10	0.35
Ge g/t	NS	110	115	-60	Z	X	Z	0.24	250	60	10	0.76	–	–	–	–
Co ppm	NS	110	115	-60	Z	X	Z	0.4	90	90	10	0.6	–	–	–	–
Cd ppm	NS	110	115	-60	Z	X	Z	0.02	30	15	14	0.61	80	55	14	0.37
Density	NS	110	115	-60	Z	X	Z	0.12	60	25	6	0.88	–	–	–	–
Big Zinc																
Cu %	NS	100	115	90	Z	X	Z	0.15	25	8	7	0.72	80	70	10	0.13
Pb %	NS	100	115	90	Z	X	Z	0.04	15	10	23	0.65	170	40	23	0.31
Zn %	NS	100	115	90	Z	X	Z	0.01	20	10	15	0.44	80	60	30	0.55
S %	NS	100	115	90	Z	X	Z	0.04	15	10	30	0.57	70	10	30	0.39
As %	NS	100	115	90	Z	X	Z	0.16	15	10	9	0.69	65	10	9	0.15
Ag g/t	NS	100	115	90	Z	X	Z	0.07	20	4	10	0.52	55	30	15	0.41
Ge g/t	NS	100	115	90	Z	X	Z	0.06	15	10	25	0.61	95	75	25	0.33
Co ppm	NS	100	115	90	Z	X	Z	0.46	30	10	11	0.21	30	35	11	0.33
Cd ppm	NS	100	115	90	Z	X	Z	0.01	10	10	10	0.4	35	35	20	0.59
Density	NS	100	115	90	Z	X	Z	0.09	10	25	22	0.55	50	50	22	0.36
Série Récurrenté																
Cu %	NS	-170	90	50	Z	X	Z	0.21	15	10	12	0.35	150	150	20	0.44
Pb %	NS	-170	90	50	Z	X	Z	0.11	100	5	15	0.34	100	75	30	0.55
Zn %	NS	-170	90	50	Z	X	Z	0.16	10	15	35	0.48	200	100	35	0.36
S %	NS	-170	90	50	Z	X	Z	0.22	30	15	7	0.35	170	125	23	0.43
As %	NS	-170	90	50	Z	X	Z	0.18	48	25	8	0.53	170	120	20	0.29
Ag g/t	NS	-170	90	50	Z	X	Z	0.34	35	50	13	0.45	100	50	13	0.21
Ge g/t	NS	-170	90	50	Z	X	Z	0.26	70	70	8	0.74	–	–	–	–
Co ppm	NS	-170	90	50	Z	X	Z	0.81	30	30	23	0.19	–	–	–	–
Cd ppm	NS	-170	90	50	Z	X	Z	0.19	10	10	6	0.58	95	65	20	0.23
Density	NS	-170	90	50	Z	X	Z	0.08	10	10	16	0.59	145	145	31	0.33

14.5.2 Indicator Variograms

The mineralization at Kipushi, in particular the Big Zinc zone, consists of extensive massive sulphide zones with pods of low grade material. It would be in-optimal to dilute the high grade massive sulphide 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. Indicator variograms were calculated using the 2 m sample composites and modelled at a threshold of 5% Zn for the zinc rich zones and 0.5% Cu for the copper rich zones.

The indicator variograms were modelled in three directions, the variogram models being robust and informed by a reasonable number of experimental data. The variograms for the Big Zinc zone were applied to the Southern Zinc zone while adjusting the direction of continuity to the strike of this zone. The variograms for the Fault Zone were applied to Zone 6 and 7 and the variograms for the Série Récurrenté zone were applied to Zone 5.

Table 14.8 Indicator Variogram Parameters – Kipushi

Attribute	**Transform**	**Rotation Angle**			**Rotation Axis**			**Nugget Effect (C^0)**	**Range of Structure 1 (R^1)**			**Sill 1 (C^1)**	**Range of Structure 2 (R^2)**			**Sill 2 (C^2)**
		1	**2**	**3**	**1**	**2**	**3**		**1**	**2**	**3**		**1**	**2**	**3**	
Fault Zone																
Cu Indicator (0.5%)	None	110	115	-60	Z	X	Z	0.23	10	25	10	0.39	100	75	10	0.38
Big Zinc																
Zinc Indicator (0.5%)	None	110	115	90	Z	X	Z	0.15	20	20	35	0.59	75	65	45	0.26
Série Récurrenté																
Cu Indicator (0.5%)	None	-170	90	50	Z	X	Z	0.39	20	15	5	0.26	135	80	8	0.35

14.6 Block Modelling

The wireframes were filled with cells 5 mX by 5 mY by 5 mZ, 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 1 mX by 1 mY by 1 mZ in order to best fill the irregular shapes of the mineralized bodies.

The seven different zone wireframes were filled separately and the blocks 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 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.

Table 14.9 Indicator Search Parameters – Kipushi

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
High–grade zone in Série Récurrente (Zone 5)																	
Cu Indicator (0.5%)	-170	90	50	Z	X	Z	80	80	40	4	8	1.5	4	4	10	4	4
Copper rich zone in Big Zinc (Zone 6)																	
Cu Indicator (0.5%)	110	90	90	Z	X	Z	145	75	10	4	8	1.5	4	4	10	4	4
Splay Zone (Zone 7)																	
Cu Indicator (0.5%)	85	90	90	Z	X	Z	75	75	10	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, estimating into parent cells. Any cells that were not estimated were assigned the domain average values for either the above or below threshold data. A maximum of four composites from a single drillhole were allowed to estimate a cell in order to ensure that each estimate was estimated by more than one drillhole.

Each cell was estimated twice; an estimate using the below threshold data and an estimate using the above threshold data. The two estimates were then combined based on the proportion of above or below threshold as determined by the indicator kriging.

The same search parameters and variograms were used to estimate the above and below threshold values. The search parameters used are shown in Table 14.10. For Zone 5, the same parameters were used as for Zone 4, and for Zone 6 and 7 the same parameters were used as for Zone 1. 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. A 52° south–west plunge direction within the plane of mineralization was modelled for Zone 1. For Zone 2, a strong down dip plunge was used based on the continuity analysis which was also applied to Zone 3. A 50° plunge to the west in the plane of mineralization was applied to Zone 4.

14.7.2.1 Boundary Conditions

Each domain was estimated only using the drillhole data within it (hard boundaries). This is with the exception of Zone 6 (the high grade copper zone in the Big Zinc) where a semi-soft boundary was used that allowed one adjacent sample composite from Zone 2, as well as the sample composites in Zone 6, to estimate the Zone 6 grade. This was based on observations on the core that found that the transition from the high grade sphalerite mineralization in Zone 3 to the high grade copper mineralization in Zone 6 was not sharp, but rather a gradual change over several metres. Likewise the Zone 2 estimate allowed for one sample within Zone 6 to be used.

Table 14.10 Search Parameters – Kipushi

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 %	110	115	-60	Z	X	Z	60	70	10	6	12	1.5	6	12	100	5	10
Pb g/t	110	115	-60	Z	X	Z	115	115	14	6	12	1.5	6	12	100	5	10
Zn %	110	115	-60	Z	X	Z	80	55	14	6	12	1.5	6	12	100	5	10
S %	110	115	-60	Z	X	Z	65	35	10	6	12	1.5	6	12	100	5	10
As %	110	115	-60	Z	X	Z	25	25	8	6	12	1.5	6	12	100	5	10
Ag g/t	110	115	-60	Z	X	Z	125	125	10	6	12	1.5	6	12	100	5	10
Ge g/t	110	115	-60	Z	X	Z	250	60	10	6	12	1.5	6	12	100	5	10
Co ppm	110	115	-60	Z	X	Z	90	90	10	6	12	1.5	6	12	100	5	10
Cd ppm	110	115	-60	Z	X	Z	80	55	14	6	12	1.5	6	12	100	5	10
Density	110	115	-60	Z	X	Z	60	25	6	6	12	1.5	6	12	100	5	10
Big Zinc (Zone 2)																	
Cu %	100	115	90	Z	X	Z	80	70	10	6	12	1.5	6	12	100	5	10
Pb g/t	100	115	90	Z	X	Z	170	40	23	6	12	1.5	6	12	100	5	10
Zn %	100	115	90	Z	X	Z	80	60	30	6	12	1.5	6	12	100	5	10
S %	100	115	90	Z	X	Z	70	10	30	6	12	1.5	6	12	100	5	10
As %	100	115	90	Z	X	Z	65	10	9	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.**
Ag g/t	100	115	90	Z	X	Z	55	30	15	6	12	1.5	6	12	100	5	10
Ge g/t	100	115	90	Z	X	Z	95	75	25	6	12	1.5	6	12	100	5	10
Co ppm	100	115	90	Z	X	Z	30	35	11	6	12	1.5	6	12	100	5	10
Cd ppm	100	115	90	Z	X	Z	80	60	30	6	12	1.5	6	12	100	5	10
Density	100	115	90	Z	X	Z	50	50	22	6	12	1.5	6	12	100	5	10
Southern Zinc (Zone 3)																	
Cu %	120	110	90	Z	X	Z	80	70	10	6	12	1.5	6	12	100	5	10
Pb g/t	120	110	90	Z	X	Z	170	40	23	6	12	1.5	6	12	100	5	10
Zn %	120	110	90	Z	X	Z	80	60	30	6	12	1.5	6	12	100	5	10
S %	120	110	90	Z	X	Z	70	10	30	6	12	1.5	6	12	100	5	10
As %	120	110	90	Z	X	Z	65	10	9	6	12	1.5	6	12	100	5	10
Ag g/t	120	110	90	Z	X	Z	55	30	15	6	12	1.5	6	12	100	5	10
Ge g/t	120	110	90	Z	X	Z	95	75	25	6	12	1.5	6	12	100	5	10
Co ppm	120	110	90	Z	X	Z	30	35	11	6	12	1.5	6	12	100	5	10
Cd ppm	120	110	90	Z	X	Z	80	60	30	6	12	1.5	6	12	100	5	10
Density	120	110	90	Z	X	Z	50	50	22	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.
Série Récurrente (Zone 4)																	
Cu %	-170	90	50	Z	X	Z	150	150	20	6	12	1.5	6	12	100	5	10
Pb g/t	-170	90	50	Z	X	Z	100	75	30	6	12	1.5	6	12	100	5	10
Zn %	-170	90	50	Z	X	Z	200	100	35	6	12	1.5	6	12	100	5	10
S %	-170	90	50	Z	X	Z	170	125	23	6	12	1.5	6	12	100	5	10
As %	-170	90	50	Z	X	Z	170	120	20	6	12	1.5	6	12	100	5	10
Ag g/t	-170	90	50	Z	X	Z	100	50	13	6	12	1.5	6	12	100	5	10
Ge g/t	-170	90	50	Z	X	Z	70	70	8	6	12	1.5	6	12	100	5	10
Co ppm	-170	90	50	Z	X	Z	30	30	23	6	12	1.5	6	12	100	5	10
Cd ppm	-170	90	50	Z	X	Z	200	100	35	6	12	1.5	6	12	100	5	10
Density	-170	90	50	Z	X	Z	145	145	31	6	12	1.5	6	12	100	5	10
High–grade zone in Série Récurrente (Zone 5)																	
Cu %	-170	90	50	Z	X	Z	150	150	20	6	12	1.5	6	12	100	5	10
Pb g/t	-170	90	50	Z	X	Z	100	75	30	6	12	1.5	6	12	100	5	10
Zn %	-170	90	50	Z	X	Z	200	100	35	6	12	1.5	6	12	100	5	10
S %	-170	90	50	Z	X	Z	170	125	23	6	12	1.5	6	12	100	5	10
As %	-170	90	50	Z	X	Z	170	120	20	6	12	1.5	6	12	100	5	10
Ag g/t	-170	90	50	Z	X	Z	100	50	13	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.
Ge g/t	-170	90	50	Z	X	Z	70	70	8	6	12	1.5	6	12	100	5	10
Co ppm	-170	90	50	Z	X	Z	30	30	23	6	12	1.5	6	12	100	5	10
Cd ppm	-170	90	50	Z	X	Z	200	100	35	6	12	1.5	6	12	100	5	10
Density	-170	90	50	Z	X	Z	145	145	31	6	12	1.5	6	12	100	5	10
Copper rich zone in Big Zinc (Zone 6)																	
Cu %	110	90	90	Z	X	Z	60	70	10	6	12	1.5	6	12	100	5	10
Pb g/t	110	90	90	Z	X	Z	115	115	14	6	12	1.5	6	12	100	5	10
Zn %	110	90	90	Z	X	Z	80	55	14	6	12	1.5	6	12	100	5	10
S %	110	90	90	Z	X	Z	65	35	10	6	12	1.5	6	12	100	5	10
As %	110	90	90	Z	X	Z	25	25	8	6	12	1.5	6	12	100	5	10
Ag g/t	110	90	90	Z	X	Z	125	125	10	6	12	1.5	6	12	100	5	10
Ge g/t	110	90	90	Z	X	Z	250	60	10	6	12	1.5	6	12	100	5	10
Co ppm	110	90	90	Z	X	Z	90	90	10	6	12	1.5	6	12	100	5	10
Cd ppm	110	90	90	Z	X	Z	80	55	14	6	12	1.5	6	12	100	5	10
Density	110	90	90	Z	X	Z	60	25	6	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.
Splay Zone (Zone 7)																	
Cu %	85	90	90	Z	X	Z	60	70	10	6	12	1.5	6	12	100	5	10
Pb g/t	85	90	90	Z	X	Z	115	115	14	6	12	1.5	6	12	100	5	10
Zn %	85	90	90	Z	X	Z	80	55	14	6	12	1.5	6	12	100	5	10
S %	85	90	90	Z	X	Z	65	35	10	6	12	1.5	6	12	100	5	10
As %	85	90	90	Z	X	Z	25	25	8	6	12	1.5	6	12	100	5	10
Ag g/t	85	90	90	Z	X	Z	125	125	10	6	12	1.5	6	12	100	5	10
Ge g/t	85	90	90	Z	X	Z	250	60	10	6	12	1.5	6	12	100	5	10
Co ppm	85	90	90	Z	X	Z	90	90	10	6	12	1.5	6	12	100	5	10
Cd ppm	85	90	90	Z	X	Z	80	55	14	6	12	1.5	6	12	100	5	10
Density	85	90	90	Z	X	Z	60	25	6	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,
- 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 and it was found that the model validated reasonably well against the data. A section showing the block model and drillholes is shown in Figure 14.11.

Figure 14.11 Section through Big Zinc and Fault Zone block model and drillhole data illustrating correlation between model and data, shaded by zinc (left) and copper (right)

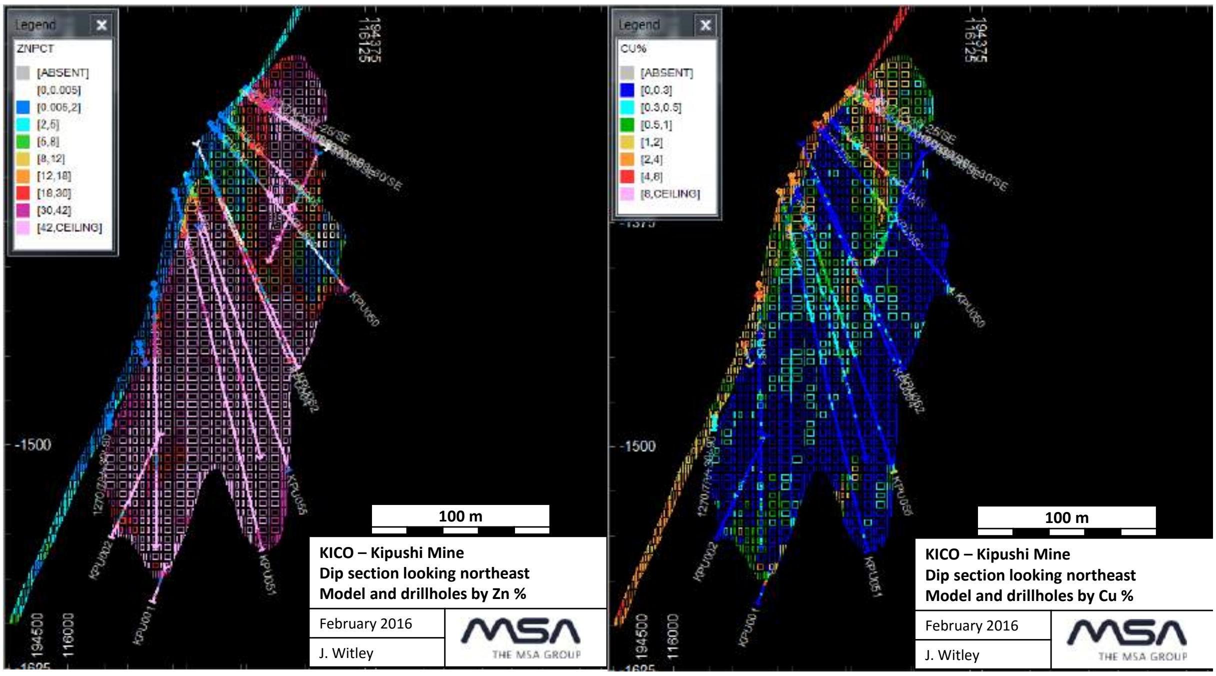
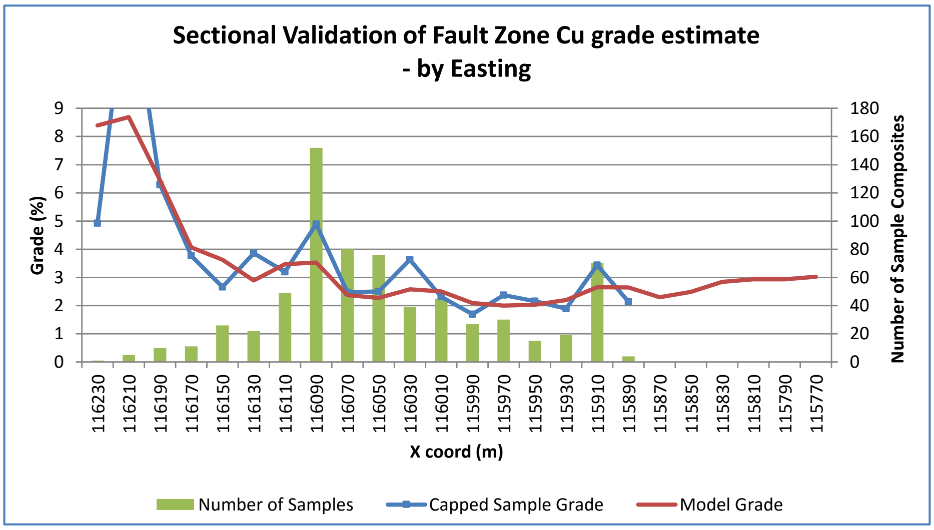
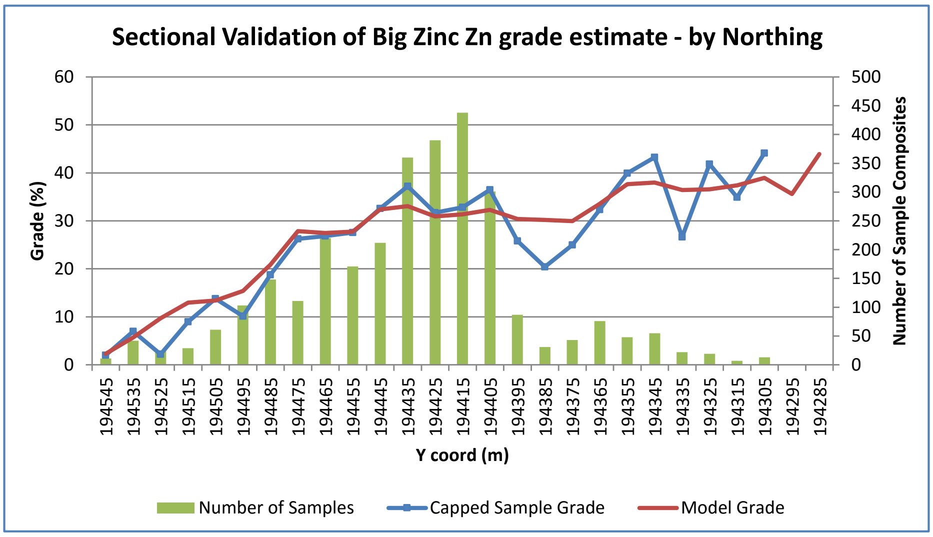


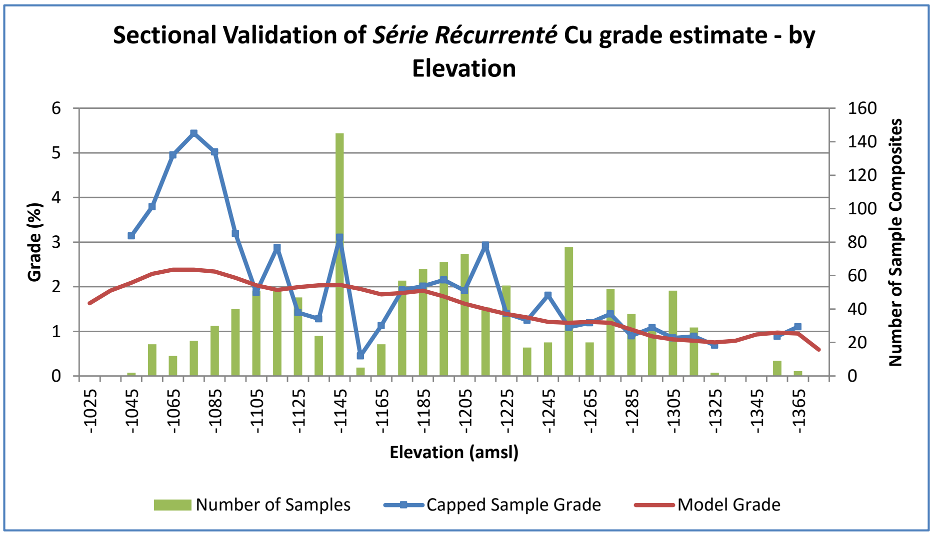
Figure by MSA, 2016.

Sectional validation plots were constructed for each major element representing 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 14.12. These show that the estimates retain the local grade trends across the deposit.

Figure 14.12 Sectional Validation Plots






Sectional Validation of *Série Récurrenté* Cu grade estimate - by Elevation

As a further check, the declustered drillhole composite mean grades were compared with the model grade (Table 14.11). The model and the data averages compare reasonably well for most variables. Those that did not compare within reasonable limits (+/-10%) 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.

- The germanium and cobalt grade of the model is significantly higher than the mean of the data for Zone 1. Only the KICO drillholes were assayed for these elements 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. This does not impact on the Mineral Resource estimate as the extrapolated area is in Zambia.
- The copper grades for the Zone 2 model are significantly lower than the mean of the data. Higher copper grades are found on the edges of the model, in the up dip area and concentrated in the apophyses which have a lower volume than the lower grade central areas.
- The arsenic grade for the Zone 3 model is significantly higher than the mean of the data. There is little arsenic data available for this zone and the data tends to occur around the edges. As the amount of data is small and the data arrangement poor the model is susceptible to the position of the few high grade values.
- Only a small portion of Zone 4 contains KICO data, there being no silver, germanium, cobalt or cadmium data in the Gécamines data. The estimate is susceptible to extrapolation of the higher and lower grade composites on the fringes of the KICO data that do not well represent the data mean.
- Large differences between the model grade and the data grade occur for several elements in Zone 5. There are a low number of composites available to estimate the grade of the zone and the model is very susceptible to the position of high or low grade samples. This portion of the Kipushi model represents only 0.3% of the total Kipushi model and does not represent a significant risk to the estimate.

- Large differences between the model grade and the data grade occur for lead and cobalt in Zone 6. The difference in the lead values is due to a cluster of high grade lead samples extrapolated into an area of the model with no lead data. A protuberance of the model is well informed by high grade cobalt values which represent a small model volume. This portion of the Kipushi model represents only 0.5% of the total Kipushi model and does not represent a significant risk to the estimate.
- Large differences between the data and model grades occur in Zone 7. This zone is informed by six drillholes with highly variable grades and is very susceptible to the data arrangement.

Table 14.11 Comparison between Drillhole and Model Data Values

Variable	Data Mean	Data Mean (Capped)	Model Mean	% Difference Model vs Capped Data
Zone 1				
Cu %	2.89	2.86	2.68	-6.3%
Pb %	0.11	0.09	0.09	-0.1%
Zn %	3.60	3.60	4.75	31.8%
S %	11.56	11.56	12.01	3.9%
As %	0.24	0.23	0.20	-13.4%
Ag g/t	18.8	18.8	16.5	-12.2%
Ge ppm	14.2	13.8	19.7	42.5%
Co ppm	193	119	205	73.0%
Cd ppm	192	187	241	28.5%
Density	3.24	3.24	3.27	1.0%
Zone 2				
Cu %	1.09	1.06	0.81	-23.8%
Pb %	0.79	0.79	0.78	-0.7%
Zn %	28.17	28.17	29.55	4.9%
S %	23.15	23.15	22.94	-0.9%
As %	0.18	0.18	0.17	-6.0%
Ag g/t	13.7	13.5	15.1	11.9%
Ge ppm	47.9	47.5	44.8	-5.8%
Co ppm	16	13	14	3.3%
Cd ppm	1318	1318	1429	8.4%
Density	3.69	3.69	3.68	-0.3%

Variable	Data Mean	Data Mean (Capped)	Model Mean	% Difference Model vs Capped Data
Zone 3				
Cu %	1.85	1.82	1.58	-13.3%
Pb %	1.35	1.35	1.58	16.9%
Zn %	17.37	17.37	17.81	2.5%
S %	21.35	21.35	21.48	0.6%
As %	0.23	0.23	0.28	21.7%
Ag g/t	–	–	12.8	–
Ge ppm	–	–	41.4	–
Co ppm	–	–	15	–
Cd ppm	831	831	858	3.3%
Density	3.58	3.58	3.59	0.4%
Zone 4				
Cu %	1.93	1.91	1.78	-6.8%
Pb %	0.04	0.03	0.02	-22.9%
Zn %	0.92	0.84	0.72	-14.7%
S %	2.89	2.89	2.50	-13.6%
As %	0.07	0.06	0.05	-9.1%
Ag g/t	8.0	8.0	8.9	11.4%
Ge ppm	0.8	0.8	0.9	23.2%
Co ppm	29	25	29	18.1%
Cd ppm	43	43	37	-13.8%
Density	3.13	3.13	3.13	-0.1%
Zone 5				
Cu %	12.99	12.99	11.94	-8.1%
Pb %	0.22	0.05	0.08	54.6%
Zn %	14.59	14.59	16.40	12.4%
S %	21.65	21.65	21.64	0.0%
As %	0.51	0.37	0.33	-10.3%
Ag g/t	58.3	54.5	56.5	3.6%
Ge ppm	20.7	20.7	22.9	10.7%
Co ppm	179	98	117	19.7%
Cd ppm	923	923	1091	18.2%

Variable	Data Mean	Data Mean (Capped)	Model Mean	% Difference Model vs Capped Data
Density	3.71	3.71	3.70	-0.2%
Zone 6				
Cu %	6.48	6.48	6.41	-1.1%
Pb %	0.77	0.77	1.02	32.5%
Zn %	25.94	25.94	23.94	-7.7%
S %	25.81	25.81	23.07	-10.6%
As %	0.20	0.20	0.18	-10.3%
Ag g/t	122.2	122.2	115.4	-5.5%
Ge ppm	61.4	61.4	61.9	0.8%
Co ppm	163	104	84	-18.9%
Cd ppm	1479	1479	1421	-3.9%
Density	3.80	3.80	3.66	-3.8%
Zone 7				
Cu %	2.99	2.99	2.75	-8.1%
Pb %	0.00	0.00	0.01	
Zn %	22.42	22.42	29.37	31.0%
S %	24.17	24.17	27.16	12.4%
As %	2.33	2.33	2.08	-10.7%
Ag g/t	14.0	14.0	14.3	1.9%
Ge ppm	125.3	125.3	173.4	38.4%
Co ppm	99	88	96	8.6%
Cd ppm	1480	1480	1899	28.3%
Density	3.71	3.71	3.81	2.7%

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 have been 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 zone and portions of the Fault Zone which yielded reasonable comparisons.

- The Gécamines data is 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.
- Large areas of the Fault Zone and Série Récurrenté zone and the entire Southern Zinc zone are only informed by Gécamines drillholes. The Big Zinc zone has been well drilled by KICO as well as a portion of the Série Récurrenté zone and Fault Zone.
- The geological framework of the Mineral Resource is well understood as are the controls to the mineralization.
- The Mineral Resource has been densely drilled on sections spaced 15 m apart, although areas of the Série Récurrenté zone 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 Mine has an extensive mining history and the continuity of the mineralized 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 mineralized 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 zone, the Fault Zone in the vicinity of the Big Zinc zone and an area of the Série Récurrenté zone. Consideration of the proximity to the areas of historic 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 all of the Southern Zinc zone and large areas of the Fault Zone and Série Récurrenté zones.
- The Splay zone 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 mineralization. Grades in this area are variable and the interpretation of the mineralized extents is tenuous.
- Extrapolation of the Big Zinc zone was limited to a maximum of 15 m, the complex shape of the deposit negated against extrapolation with any confidence. The Fault Zone and Série Récurrenté zone are highly continuous and the down dip extent was limited to 50 m from the drillhole intersections.

The classified areas are shown in Figure 14.13 for the Big Zinc zone, Figure 14.14, for the Fault Zone and Figure 14.15 for the Série Récurrenté zone.

To the best of the Qualified Person's knowledge there are 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 2016 PEA, aside from those mentioned in Section 4.

The Mineral Resources will 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 14.13 Mineral Resource Classification, Big Zinc – Isometric View Looking Approximately East

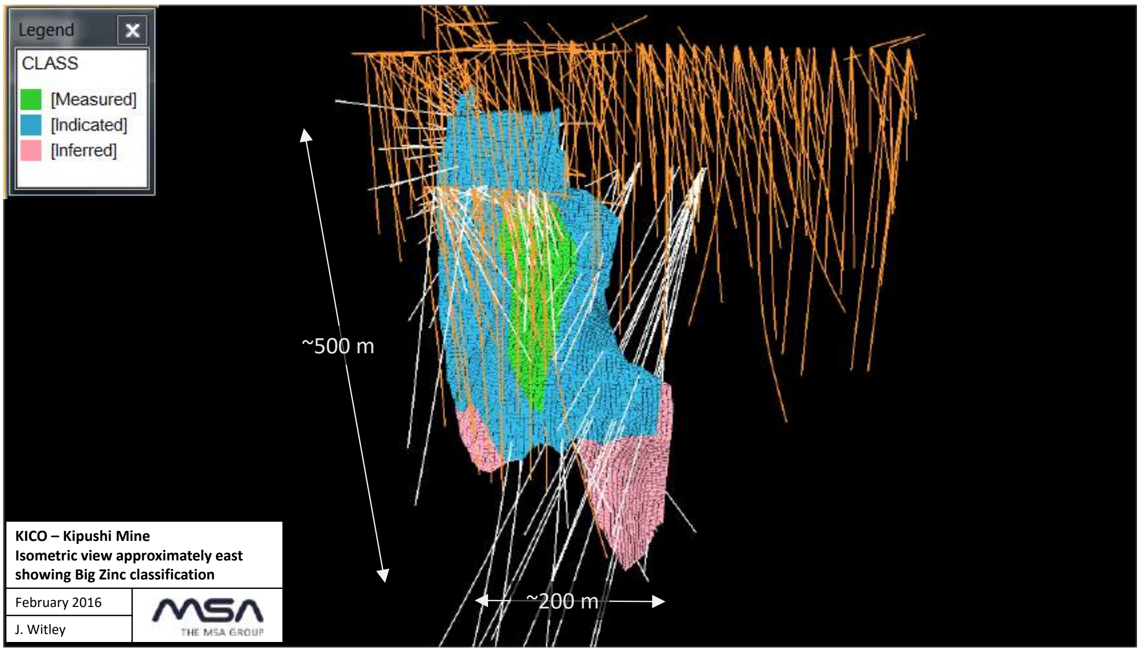


Figure by MSA, 2016.

White traces = KICO drillholes

Orange traces = Gécamines drillholes

Figure 14.14 Mineral Resource Classification, Fault Zone – Isometric View Looking Approximately North–west

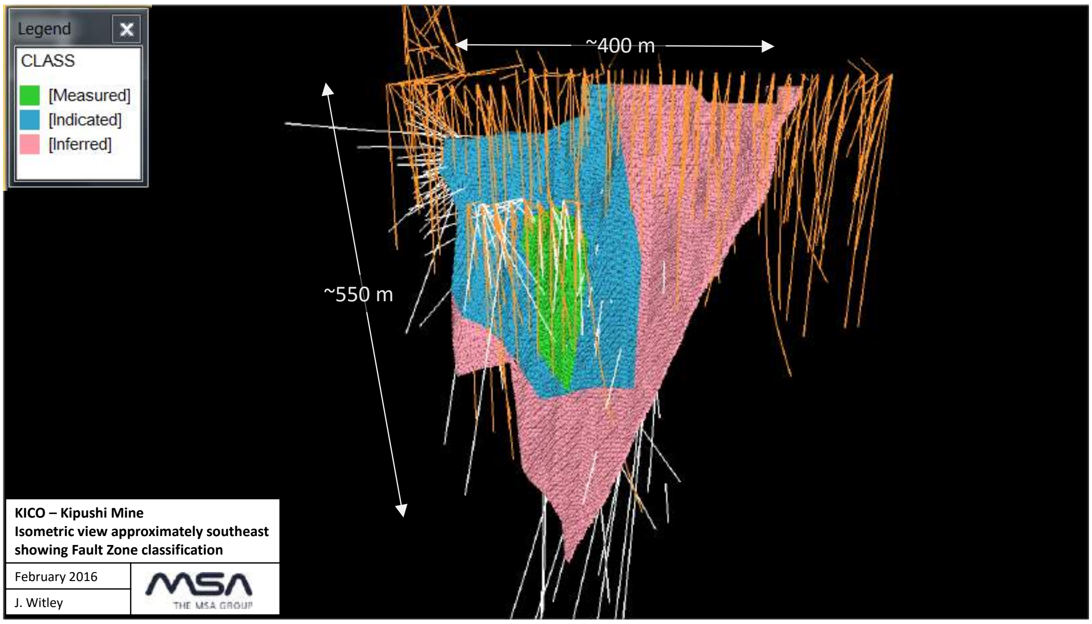


Figure by MSA, 2016.

White traces = KICO drillholes

Orange traces = Gécamines drillholes

Figure 14.15 Mineral Resource Classification, Série Récurrenté – Isometric View Looking Approximately South–east

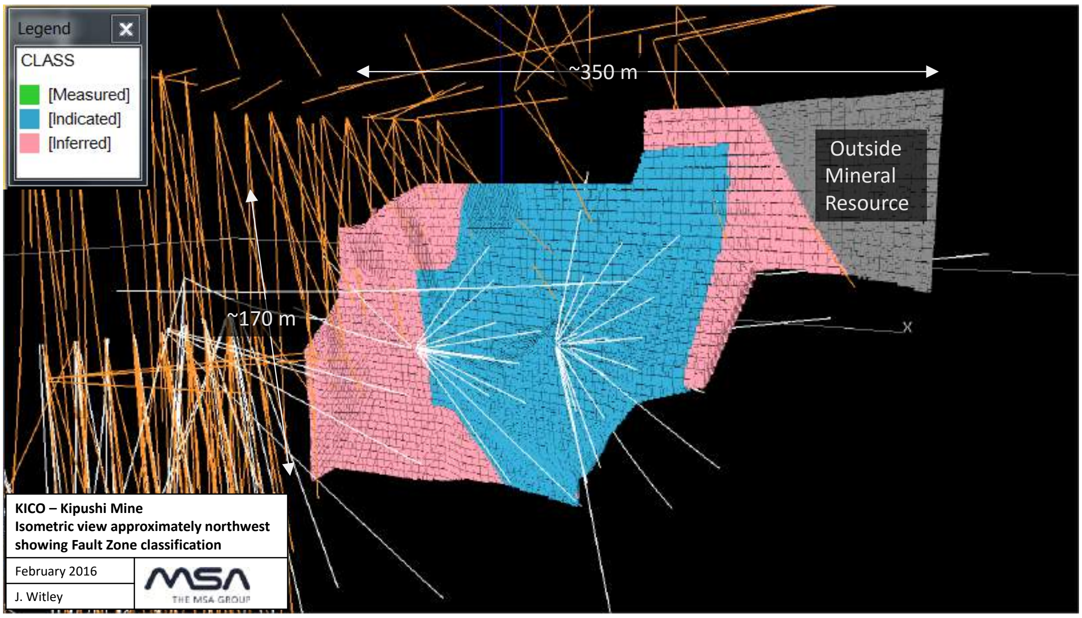


Figure by MSA, 2016.

White traces = KICO drillholes

Orange traces = Gécamines drillholes

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, all of the model above 1,150 mRL was removed, extensive mining having taken place in the levels above. There is potential for additional Mineral Resources to exist above 1,150 mRL but this will require investigation in terms of mineralization remaining and reasonable prospects for economic extraction of the remnant areas.

14.10.2 Zambia-DRC Border

The mineralization at Kipushi straddles the DRC-Zambia border, however the exact location of the position of the border is uncertain at Kipushi, there being no officially 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 Tswane, 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 14.16). 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 14.16 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 mineralization models and all modelled mineralization on the Zambian side of the border line was discounted 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 14.12 for the predominantly zinc-rich bodies and in Table 14.13 for the predominantly copper-rich bodies.

The Measured and Indicated, and Inferred Mineral Resource for the zinc-rich bodies has been tabulated using a number of cut-off grades as shown in Table 14.14 and Table 14.15 respectively and Table 14.16 and Table 14.17 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 mineralization 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, the level of study so far carried out being insufficient with which to do so.

Table 14.12 Kipushi Zinc-Rich Mineral Resource at 7% Zn Cut-off Grade, 23 January 2016

Zone	Category	Tonnes (Millions)	Zn (%)	Cu (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
Big Zinc	Measured	3.59	38.39	0.67	0.36	18	17	54
	Indicated	6.60	32.99	0.63	1.29	20	14	50
	Inferred	0.98	36.96	0.79	0.14	7	16	62
Southern Zinc	Indicated	0.00	–	–	–	–	–	–
	Inferred	0.89	18.70	1.61	1.70	13	15	43
Total	**Measured**	**3.59**	**38.39**	**0.67**	**0.36**	**18**	**17**	**54**
	Indicated	**6.60**	**32.99**	**0.63**	**1.29**	**20**	**14**	**50**
	Measured & Indicated	**10.18**	**34.89**	**0.65**	**0.96**	**19**	**15**	**51**
	Inferred	**1.87**	**28.24**	**1.18**	**0.88**	**10**	**15**	**53**

		Contained Metal Quantities						
Zone	**Category**	**Tonnes (Millions)**	**Zn Pounds (Millions)**	**Cu Pounds (Millions)**	**Pb Pounds (Millions)**	**Ag Ounces (Millions)**	**Co Pounds (Millions)**	**Ge Ounces (Millions)**
Big Zinc	Measured	3.59	3,035.8	53.1	28.7	2.08	0.13	6.18
	Indicated	6.60	4,797.4	91.9	187.7	4.15	0.20	10.54
	Inferred	0.98	797.2	17.1	3.0	0.23	0.03	1.96
Southern Zinc	Indicated	0.00	0.0	0.0	0.0	0.00	0.00	0.00
	Inferred	0.89	368.6	31.8	33.5	0.38	0.03	1.23
Total	**Measured**	**3.59**	**3,035.8**	**53.1**	**28.7**	**2.08**	**0.13**	**6.18**
	Indicated	**6.60**	**4,797.4**	**91.9**	**187.7**	**4.15**	**0.20**	**10.54**
	Measured & Indicated	**10.18**	**7,833.3**	**144.9**	**216.4**	**6.22**	**0.33**	**16.71**
	Inferred	**1.87**	**1,168.7**	**49.6**	**36.8**	**0.61**	**0.06**	**3.21**

Notes:

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.
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 $1.02/lb, mining cost of $50 /tonne, processing cost of $10/tonne, G&A and holding cost of $10 /tonne, transport of 55% Zn concentrate at $375/tonne, 90% zinc recovery and 85% payable zinc.

Table 14.13 Kipushi Copper-Rich Mineral Resource at 1.5% Cu Cut-off grade, 23 January 2016

Zone	Category	Tonnes (Millions)	Cu (%)	Zn (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
Fault Zone	Measured	0.14	2.78	1.25	0.05	19	107	20
	Indicated	1.01	4.17	2.64	0.09	23	216	20
	Inferred	0.94	2.94	5.81	0.18	22	112	26
Série Récurrenté	Indicated	0.48	4.01	3.82	0.02	21	56	6
	Inferred	0.34	2.57	1.02	0.06	8	29	1
Fault Zone Splay	Inferred	0.35	4.99	15.81	0.005	20	127	81
Total	**Measured**	**0.14**	**2.78**	**1.25**	**0.05**	**19**	**107**	**20**
	Indicated	**1.49**	**4.12**	**3.02**	**0.07**	**22**	**165**	**15**
	Measured & Indicated	**1.63**	**4.01**	**2.87**	**0.06**	**22**	**160**	**16**
	Inferred	**1.64**	**3.30**	**6.97**	**0.12**	**19**	**98**	**33**

		Contained Metal Quantities						
Zone	**Category**	**Tonnes (Millions)**	**Cu Pounds (Millions)**	**Zn Pounds (Millions)**	**Pb Pounds (Millions)**	**Ag Ounces (Millions)**	**Co Pounds (Millions)**	**Ge Ounces (Millions)**
Fault Zone	Measured	0.14	8.5	3.8	0.2	0.09	0.03	0.09
	Indicated	1.01	93.2	59.1	1.9	0.75	0.48	0.64
	Inferred	0.94	61.1	120.9	3.8	0.68	0.23	0.79
Série Récurrenté	Indicated	0.48	42.4	40.5	0.2	0.32	0.06	0.09
	Inferred	0.34	19.4	7.7	0.4	0.09	0.02	0.01
Fault Zone Splay	Inferred	0.35	38.9	123.3	0.0	0.23	0.10	0.92
Total	**Measured**	**0.14**	**8.5**	**3.8**	**0.2**	**0.09**	**0.03**	**0.09**
	Indicated	**1.49**	**135.7**	**99.6**	**2.1**	**1.08**	**0.54**	**0.73**
	Measured & Indicated	**1.63**	**144.1**	**103.4**	**2.3**	**1.16**	**0.58**	**0.82**
	Inferred	**1.64**	**119.4**	**251.8**	**4.3**	**1.00**	**0.35**	**1.73**

Notes:

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.
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 $2.97/lb, mining cost of $50/tonne, processing cost of $10/tonne, G&A and holding cost of $10/tonne, 90% copper recovery and 96% payable copper.

Table 14.14 Kipushi Zinc-Rich Bodies Measured and Indicated Mineral Resource Grade Tonnage Table, 23 January 2016

Cut-Off (Zn %)	Tonnes (Millions)	Zn (%)	Zn Pounds (Millions)	Cu (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
5	10.46	34.12	7,870.0	0.65	0.95	19	15	50
7	**10.18**	**34.89**	**7,833.3**	**0.65**	**0.96**	**19**	**15**	**51**
10	9.78	35.99	7,757.4	0.63	0.98	19	15	52
12	9.50	36.72	7,689.4	0.62	1.00	19	15	53
15	9.06	37.85	7,559.1	0.59	1.01	20	15	54

Notes:

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.
4. Metal quantities are reported in multiples of Troy Ounces or Avoirdupois Pounds.

Table 14.15 Kipushi Zinc-Rich Bodies Inferred Mineral Resource Grade Tonnage Table, 23 January 2016

Cut-Off (Zn %)	Tonnes (Millions)	Zn (%)	Zn Pounds (Millions)	Cu (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
5	1.89	27.98	1,168.8	1.19	0.88	10	15	53
7	**1.87**	**28.24**	**1,165.7**	**1.18**	**0.88**	**10**	**15**	**53**
10	1.82	28.85	1,154.8	1.17	0.88	10	15	54
12	1.75	29.47	1,139.8	1.15	0.87	10	15	55
15	1.56	31.42	1,082.1	1.08	0.83	10	15	57

Notes:

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.
4. Metal quantities are reported in multiples of Troy Ounces or Avoirdupois Pounds.

Table 14.16 Kipushi Copper-Rich Bodies Indicated Mineral Resource Grade Tonnage Table, 23 January 2016

Cut-Off (Cu %)	Tonnes (Millions)	Cu (%)	Cu Pounds (Millions)	Zn (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
1.0	2.56	3.00	169.2	2.01	0.05	17	114	11
1.5	**1.63**	**4.01**	**144.1**	**2.87**	**0.06**	**22**	**160**	**16**
2.0	1.17	4.92	126.6	3.66	0.08	26	202	19
2.5	0.95	5.54	115.8	4.06	0.08	29	227	20
3.0	0.82	5.99	108.0	4.32	0.08	30	244	20

Notes:

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.
4. Metal quantities are reported in multiples of Troy Ounces or Avoirdupois Pounds.

Table 14.17 Kipushi Copper-Rich Bodies Inferred Mineral Resource Grade Tonnage Table, 23 January 2016

Cut-Off (Cu %)	Tonnes (Millions)	Cu (%)	Cu Pounds (Millions)	Zn (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
1.0	2.40	2.64	139.8	5.85	0.09	16	79	29
1.5	**1.64**	**3.30**	**119.4**	**6.97**	**0.12**	**19**	**98**	**33**
2.0	1.24	3.81	104.2	7.29	0.13	20	109	33
2.5	0.90	4.40	87.6	8.01	0.13	21	113	34
3.0	0.68	4.95	74.0	8.38	0.15	21	118	34

Notes:

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.
4. Metal quantities are reported in multiples of Troy Ounces or Avoirdupois Pounds.

The Mineral Resource was limited to deeper than approximately 1,150 mRL, extensive mining having taken place in the levels above. Below 1,150 mRL, 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 mRL is dictated by the location of the diamond drilling data, although sparse drilling completed by KICO below this elevation indicates that the mineralization 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 27 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), is a member in good standing 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 mineralization and activity being undertaken as defined in National Instrument 43-101 Standards of Disclosure of Mineral Projects.

14.12 Comparison with Previous Estimates

The Mineral Resource estimate reported as at 23 January 2016 is the first Mineral Resource for Kipushi reported in accordance with CIM.

The reader is cautioned that a Qualified Person has not done sufficient work to classify the Historical Estimate as current Mineral Resources and the issuer is not treating the Historical Estimate as current Mineral Resources. The Historical Estimate should be regarded as no longer relevant, it having been superseded by the 23 January 2016 Mineral Resource. The Historical Estimate was prepared by Techpro in accordance with the 1996 edition of the JORC Code but would not meet current JORC or CIM standards.

A summary of the Historical Estimate is shown in Table 14.18. The South and North zones together represent the copper rich zones. MSA considers that the South zone is approximately equivalent to the Fault Zone and the North zone is approximately equivalent to the Série Récurrenté zone.

Table 14.18 Summary of Kipushi Historical Estimate (Techpro 1997)

Category	Level (mRL)	South			North			Big Zinc		
		Tonnes (millions)	Cu (%)	Zn (%)	Tonnes (millions)	Cu (%)	Zn (%)	Tonnes (millions)	Cu (%)	Zn (%)
Measured	100 to 1,150	–	–	–	3.7	2.01	2.05	–	–	–
Measured	1,150 to 1,295	2.5	2.47	18.58	1.9	4.19	4.35	0.8	1.16	33.52
Indicated	1,295 to 1,500	1.5	2.27	17.04	2.6	4.09	5.25	3.9	0.68	39.57
Total M&I	**1,150 to 1,500**	**4.0**	**2.40**	**18.00**	**4.5**	**4.13**	**4.87**	**4.7**	**0.76**	**38.54**

The total Measured and Indicated Historical Estimate for the combined South and North areas is 8.5 Mt at 3.32% Cu and 11.04% Zn, excluding the area from 100–1,150 mRL. The tonnage is more than double that of the 23 January 2016 Mineral Resource estimate and the copper grade is approximately 9% lower, assuming a 1.5% Cu cut-off grade. In contrast, the Big Zinc Historical Estimate is approximately one third of that of the 23 January 2016 Mineral Resource estimate and 10% higher in grade, assuming a 7% Zn cut-off grade.

Significant differences between Techpro's Historical Estimate and the 23 January 2016 Mineral Resource estimate are explained as follows:

- A portion of the Historical Estimate classified as Measured by Techpro (3.5 Mt at 2.01% Cu and 2.05% Zn) occurs from 100–1,150 mRL. This area was not included in the 23 January 2016 Mineral Resource estimate as extensive mining has taken place in these areas and it is uncertain whether this material can be accessed for extraction.
- The Historical Estimate may have included material that is now considered to be outside of the DRC and within Zambia.
- The definition of the zinc-rich and copper-rich zones is likely to be different between the two estimates.
- The Historical Estimate was based on the results of Gécamines drilling whereas the 23 January 2016 Mineral Resource estimate used Gécamines drilling data, where appropriate, combined with significant amount of KICO drilling data completed since then. Differences in estimates using different datasets will occur.
- The extent of the Big Zinc zone has been expanded based on the KICO drilling that intersected mineralisation outside of the area of the Historical Estimate.
- The Techpro Historical Estimate was based on estimations by Gécamines that used outdated sectional interpretation methods, rather than the more modern geostatistical estimation techniques used for this Mineral Resource estimate.
- The Historical Estimate is based on the Gécamines estimate which applied 1970s metal prices which were not changed thereafter. Ground having less than 1% Cu and 7% Zn was considered to be sterile, however no precise cut-off grades were applied.

14.13 Assessment of Reporting Criteria

The checklist in Table 14.19 of assessment and reporting criteria summarises the pertinent criteria for this Mineral Resource estimate in accordance with CIM guidelines and MSA's assessment and comment on the estimates.

Table 14.19 Checklist of Reporting Criteria

Drilling techniques	All drillholes were diamond drill cored and drilled from underground (mostly NQ) at various inclinations. The drillholes were generally drilled along section lines spaced 15 m apart. The KICO drilling was largely inclined downwards at various orientations designed to intersect specific targets. Gécamines drillholes that were drilled in a similar orientation as the plane of mineralization were not used for grade estimation as samples from these holes would not be considered representative.
Logging	All of the drillholes were geologically logged by qualified geologists. The logging was of an appropriate standard for Mineral Resource estimation.
Drill sample recovery	Core recovery was observed to be excellent for the KICO drilling. The Gécamines drillhole cores were in various conditions having been stored for long periods of time.
Sampling methods	Half core samples were collected continuously through the mineralized zones after being cut longitudinally in half using a diamond saw. The KICO drillhole samples were taken at nominal 1 m intervals, which were adjusted to smaller intervals in order to honour the mineralization styles and lithological contacts. From KPU051 onwards the nominal sample interval was adjusted to 2 m intervals which were adjusted to smaller intervals in order to honour the mineralization styles and lithological contacts. MSA's observations indicated that the routine sampling methods applied by KICO were of a high standard and suitable for evaluation purposes. Sampling by Gécamines was selective and lower grade portions of the mineralized intersections were not always sampled. Sample lengths were based on homogenous zones of mineralization and varied from less than 1 m to greater than 10 m. Gécamines drillholes were not used for grade estimation where well mineralized sample lengths were considered to be excessive.
Quality of assay data and laboratory tests	All sample preparation was completed by staff from KICO and its affiliated companies at its own laboratories. From 1 June to 31 December 2014, samples were prepared at Kolwezi by staff from the company's exploration division. From January to November 2015, samples were prepared at Kamoa by staff from that project. Mr M Robertson from MSA inspected KICO's preparation facilities in the DRC. Representative pulverised subsamples were all assayed at the Bureau Veritas Minerals (BVM) laboratory in Perth, Australia. Samples were dried at between 100°C and 105°C and crushed to a nominal 70% passing 2 mm, 800 g to 1000 g subsamples were taken by riffle split, and the subsamples were milled to 90% passing 75 µm. Crushers and pulverisers were flushed with barren quartz material after each sample. Grain size monitoring tests were conducted on samples labelled duplicates, which comprise about 5% of total samples. 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; and a 90 g sample for office archives. Approximately 5% of the sample batches sent to BVM were comprised of certified reference materials, 5% of blanks and 5% crushed reject duplicates. The CRMs were certified for Zn, Cu, Pb and Ag and no CRMs were used to monitor the accuracy of As, Cd, Co, and Ge. BVM conducted 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 SPF with an ICP-MS finish; Ag and Hg Aqua Regia digestion assays with an ICP-MS finish; and Au, Pt, and Pd by lead collection fire assay with an ICP-OES finish. For Ag, Aqua Regia digest values were used below approximately 50 ppm and SPF values were used above approximately 50 ppm. A variety of certified reference materials as well as blanks and duplicates were routinely inserted and assayed by BVM as part of its own internal QAQC processes. The QAQC measures used by KICO revealed the following: • The certified reference materials demonstrated that the assays for Zn, Cu and Pb were overall unbiased. Where CRM failures were identified, the CRM and a group of samples before it and after it were submitted for re-assaying of the failed elements in most cases. Silver values reported by BVM tended to be lower than the certified mean by between approximately 2% and 15% on average for the individual CRMs.

	- Blank samples indicated that no significant contamination occurred for most of the programme. Blank results from the earlier part of the exploration programme showed more elevated concentrations than ideal, however most of the failures are in the several hundred ppm range and are well below cut-off grades that may be considered for this mineralization. - Duplicate precision levels are within reasonably expected ranges. A check assay programme was carried out by KICO. This consisted of re-assaying of 210 samples for Zn, Cu Pb, Ag, S, As, Cd, Co, Au, Hg, Ge, and Re from KPU01 to KPU025 at Genalysis (Perth) and SGS (Perth). Both laboratories validated the BVM assays within reasonable limits. Historical sampling and assaying was carried out by Gécamines at the Kipushi laboratory. Sample analysis was carried out by a four-acid digest and AAS finish for Cu, Co, Zn, and Fe. The GBC Avanta AAS instrument originally used for the assays is still operational. Sulphur analysis was carried out by the "classical" gravimetric method. No information is available on the QAQC measures implemented for the Gécamines samples and therefore the Gécamines sample assays should be considered less reliable than the KICO sample assays.
Verification of sampling and assaying	MSA observed the mineralization in the cores and compared it with the assay results. MSA found that the assays generally agreed with the observations made on the core. A re-sampling exercise of eight Gécamines drillholes was completed by KICO in 2013 under MSA's direction, and included QAQC protocols. The samples were sent to BVM for analysis. The results revealed that Gécamines Zn and Cu assays compared reasonably well overall with the BVM assays. Ten of the Gécamines holes were verified by KICO twin drilling. The Zn, Cu, and Pb values compared well overall between the twin drilling and original holes.
Location of data points	All of the KICO drillhole collars have been surveyed. Downhole surveys were completed for all of the KICO holes. The method of location for the Gécamines drillhole collars is uncertain and not all of the holes were surveyed down-the-hole.
Tonnage factors (in-situ bulk densities)	Specific gravity determinations were made for the KICO drillhole samples using the Archimedes principal of weight in air versus weight in water. A regression formula was developed using metal grades to apply density to the samples based on the KICO measurements.
Data density and distribution	The drillholes were drilled along section lines spaced 15 m apart. Along the section lines the drillholes intersected the mineralization between 10 m and 50 m apart in the Big Zinc zone and adjacent Fault Zone Mineral Resource area, with drilling being sparser, up to approximately 100 m apart, in the deeper parts of the Fault Zone. The Série Récurrenté zone was drilled along 15 m spaced lines by Gécamines with drillhole intersections approximately 50 m apart. KICO completed a number of drillhole fans over a portion of the Série Récurrenté zone, which resulted in intersections approximately 20 m apart. The number of drillhole intersections used to estimate each zone is as follows: - Fault Zone: 122 of which 45 were drilled by KICO. - Big Zinc: 100 of which 51 were drilled by KICO. - Southern Zinc: 26 of which none were drilled by KICO. - Série Récurrenté: 57 of which 32 were drilled by KICO. - Fault Zone Splay: 6 of which all were drilled by KICO. These were sourced from 107 Gécamines holes that intersected the mineralized zones and were accepted for the estimate and 84 KICO drillholes from the series KPU001 to KPU097. The Gécamines holes were not assayed for Ag, Ge, and Co.
Database integrity	The KICO data were stored in an Access database. MSA compiled a digital database of the Gécamines hard copy data.
Dimensions	The Fault Zone forms a steeply dipping irregular tabular body of variable thickness. The area defined as a Mineral Resource is approximately 420 m in strike in the up-dip areas and tapers off at depth due to the limited amount of drilling. The thickness varies from approximately 1 m to more than 20 m with typical thicknesses being between 5 m and 10 m.

	The Big Zinc zone Mineral Resource is an irregular pipe-like body elongated vertically and along the Fault Zone strike direction. It extends for a maximum of approximately 220 m along strike, 100 m in plan thickness and extends over 600 m down-dip with a steep southerly plunge. The Southern Zinc zone is elongate in the alignment of the Fault Zone and extends for approximately 200 m in strike and dip and is typically between 5 m and 15 m wide. The Série Récurrenté zone extends along strike for approximately 250 m, 300 m in the dip direction and is between 20 m and 70 m wide. The Fault Zone Splay is an irregular steeply dipping body that extends along strike for approximately 60 m, 250 m in the dip direction and is between 4 m and 20 m wide.
Geological interpretation	The mineralized intersections in drill core are clearly discernible. Three dimensional wireframe models were created for the zones of mineralization based on a grade threshold of 1.0% Cu or 5% Zn. The grade shells were aligned with the geological understanding of the mineralization trends. The mineralization is a result of large scale replacement of dolomitic horizons by hydrothermal fluids, and as a result the model boundaries are irregular.
Domains	Seven domains were created: • Fault Zone • Big Zinc • A copper-silver rich zone within the Big Zinc • Southern Zinc • Série Récurrenté • A high grade (>5%) copper-rich zone within the Série Récurrenté • Fault Zone Splay– high grade copper-zinc-germanium
Compositing	Sample lengths were composited to 2 m. All sample lengths were retained in the compositing process so that the majority of composites were close to 2 m long, however composites as narrow as 0.70 m and as wide as 2.81 m occur. There is no relationship between composite length and grade.
Statistics and variography	Copper distributions are positively skewed with coefficients of variation (CV) being approximately 1.4 for both of the Cu-rich zones. Zinc distributions in the Zn-rich zones are not skewed and grades are distributed evenly across the grade ranges. The CV is approximately 0.8 for both zones. Cadmium behaves similarly to zinc and there is a strong relationship between the two metals. Lead, germanium, silver and cobalt distributions are positively skewed with high CVs. Sulphur and density distributions are similar to those of copper and zinc in their respective zones. Missing sulphur and density values were applied to the drillholes based on regression formulae using copper plus zinc plus lead grades for each zone. A regression formula for missing cadmium values was based on its strong relationship with zinc. Normal Scores variograms were calculated in the plane of the mineralization, downhole and across strike. Variogram ranges differ widely between elements. The variogram models for zinc and cadmium are similar there being a strong relationship between these elements. For the Fault Zone, the copper variogram has a range of 60 m on strike, 70 m in the plunge direction and 10 m across strike. The zinc variogram for Big Zinc zone has a range of 80 m down dip, 60 m along strike and 30 m across dip. The copper variogram for Série Récurrenté zone has a range of 150 m in the plane of mineralization and 20 m across strike. There were insufficient data to create variograms for the Southern Zinc zone and so the Big Zinc zone variogram was applied with some modifications for the orientation of this zone. High grade copper zones were assigned variogram parameters from zones of similar grade.
Top or bottom cuts for grades	Top cuts were sparingly applied to outlier values that were above breaks in the cumulative probability plot.

<table>
<tr><td>Data clustering</td><td>Although the data are irregularly distributed there is no preferential clustering in the higher grade areas.</td></tr>
<tr><td>Block size</td><td>Block models of 5 m N by 5 m E by 5 m RL were created with a minimum sub-cell of 1 m.</td></tr>
<tr><td>Grade estimation</td><td>Grades were estimated using Ordinary Kriging into parent cells. Indicators were used to distinguish between zones of internal waste within the mineralized zone. The indicator thresholds used were 0.5% for Cu and 5.0% for Zn.
A minimum number of 6 and a maximum of 12 two metre composites were required in each of the above and below threshold populations for each variable to be estimated. Search distances were set at the respective variogram range and increased by 1.5 times the variogram range should enough samples not be collected for estimation by the first search. A further expanded search that collected a minimum of 5 and maximum of 10 samples was used to ensure that the entire model was estimated. A maximum of 4 samples were allowed to estimate a block from a single hole.
There were no silver, germanium or cobalt data available in the Southern-Zinc zone. The average values of the Big Zinc zone were applied and therefore these estimates are considered to be of low confidence.</td></tr>
<tr><td>Resource classification</td><td>The drill spacing over much of the area is sufficient to estimate grades and model the geological framework to a high degree of confidence. There is high confidence in the accuracy and integrity of the KICO data. The Gécamines data was collected using protocols that are not considered optimal today and despite reasonable validation through re-sampling and twin drilling the Gécamines data should be considered to be of low confidence. On this basis the Mineral Resource was classified as Indicated when the drillhole spacing is generally closer than a 50 m grid in the plane of mineralization and predominantly informed by KICO drilling data, while considering its location relative to the mined out areas. The Mineral Resource was classified as Inferred when informed by Gécamines drilling data even when the drilling grid was less than 50 m in order to reflect the lower confidence in this data. Where the confidence in the geological interpretation of the deposit is high and the model is informed by KICO drillholes at a spacing of approximately 20 m the Mineral Resource was classified as Measured.
The Big Zinc body is complex in shape and pinches out rapidly in areas. For this reason extrapolation of the Mineral Resource was limited to less than 15 m away from the drillhole grid. The copper zones exhibit stronger geological continuity and down-dip extrapolation was limited to a maximum of 50 m.</td></tr>
<tr><td>Mining cuts</td><td>No mining cuts were considered in the estimate. The dimension and shape of the mineralization makes it amenable to a variety of well-established mining methods.</td></tr>
<tr><td>Metallurgical factors or assumptions</td><td>The mineralization is in sulphide form and amenable to flotation. The grades of deleterious elements were estimated as follows.
<table>
<tr><td>Zinc Rich Zones
(Zn cut-off-grade 7%)</td><td>Arsenic (%)</td><td>Cadmium (ppm)</td></tr>
<tr><td>Measured and Indicated</td><td>0.17</td><td>1725</td></tr>
<tr><td>Inferred</td><td>0.27</td><td>1169</td></tr>
<tr><td>Copper Rich Zones
Cu cut-off-grade 1.5%)</td><td>Arsenic (%)</td><td>Cadmium (ppm)</td></tr>
<tr><td>Measured and Indicated</td><td>0.36</td><td>164</td></tr>
<tr><td>Inferred</td><td>0.78</td><td>339</td></tr>
</table></td></tr>
<tr><td>Legal aspects and tenure</td><td>Kipushi Corporation Sprl (KICO) is a Joint Venture between Gécamines and Ivanhoe established for the exploration, development and production of Kipushi. Exploitation permit (<i>Permis d'Exploitation</i> 12434) grants KICO the right to mine and process copper, cobalt, zinc, silver, lead and germanium from the Kipushi Zn-Cu Project until 03 April 2024.</td></tr>
</table>

Audits, reviews and site inspection	The following review work was completed by MSA: • Mike Robertson of the MSA Group visited the project from 20 February 2013 to 22 February 2013 and from 22 April 2013 to 24 April 2013. The Gécamines cores were examined and the sampling and logging records were verified against the cores. A check sampling exercise was initiated under supervision. • Jeremy Witley of the MSA Group and the Qualified Person for this Mineral Resource estimate visited the project from 8 to 11 September 2014 and 11 to 13 May 2015.

15 MINERAL RESERVE ESTIMATES

No Mineral Reserve estimates have been undertaken.

16 MINING METHODS

16.1 Mining

The planned mining method is a combination of Sublevel Open Stoping (SLOS), Pillar Retreat, and Cut and Fill methods at a steady-state mining rate of 1.1 Mtpa. The Big Zinc zone primary mining method is expected to be SLOS, with cemented rock backfill. The crown pillars are expected to be mined using Pillar Retreat mining method once the adjacent stopes are backfilled. The Copper zone outside of the Big Zinc zone has been identified to be mined by the Cut and Fill mining method.

16.2 Existing 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 2016 PEA planning are:

- Shaft 5 (0–1,240 mRL): Personnel, material, services, rock hoisting, and ventilation
- P1 Tertiary (1,115–1,485 mRL): Internal backfill pass
- Shaft 2 (0–500 mRL): Personnel, material
- P2 Bis (500–850 mRL): Second egress, personnel, materials
- Shaft 3 (0–710 mRL): Backfill pass from the surface, second egress
- Shaft 4 (0–650 mRL): Ventilation
- Shaft 4 Bis (650–825 mRL): Return ventilation
- Shaft 15 (850–1,172 mRL): Personnel, services
- Shaft 19 (825–1,120 mRL): Return ventilation

A 5 m high by 5.8 m wide decline was developed from 725 mRL to approximately 1,330 mRL, the upper to deeper working levels and the top of the Big Zinc zone.

The main working area is connected to Shaft 5 via the 1,150 mRL main haulage level. There is a crusher chamber and loading pocket at 1,200 mRL; the crusher level is now dewatered. The underground infrastructure, including the crusher, exposed since dewatering, is in relatively good order.

A schematic layout of the existing development is shown in Figure 16.1.

Figure 16.1 Schematic Section of Kipushi Mine

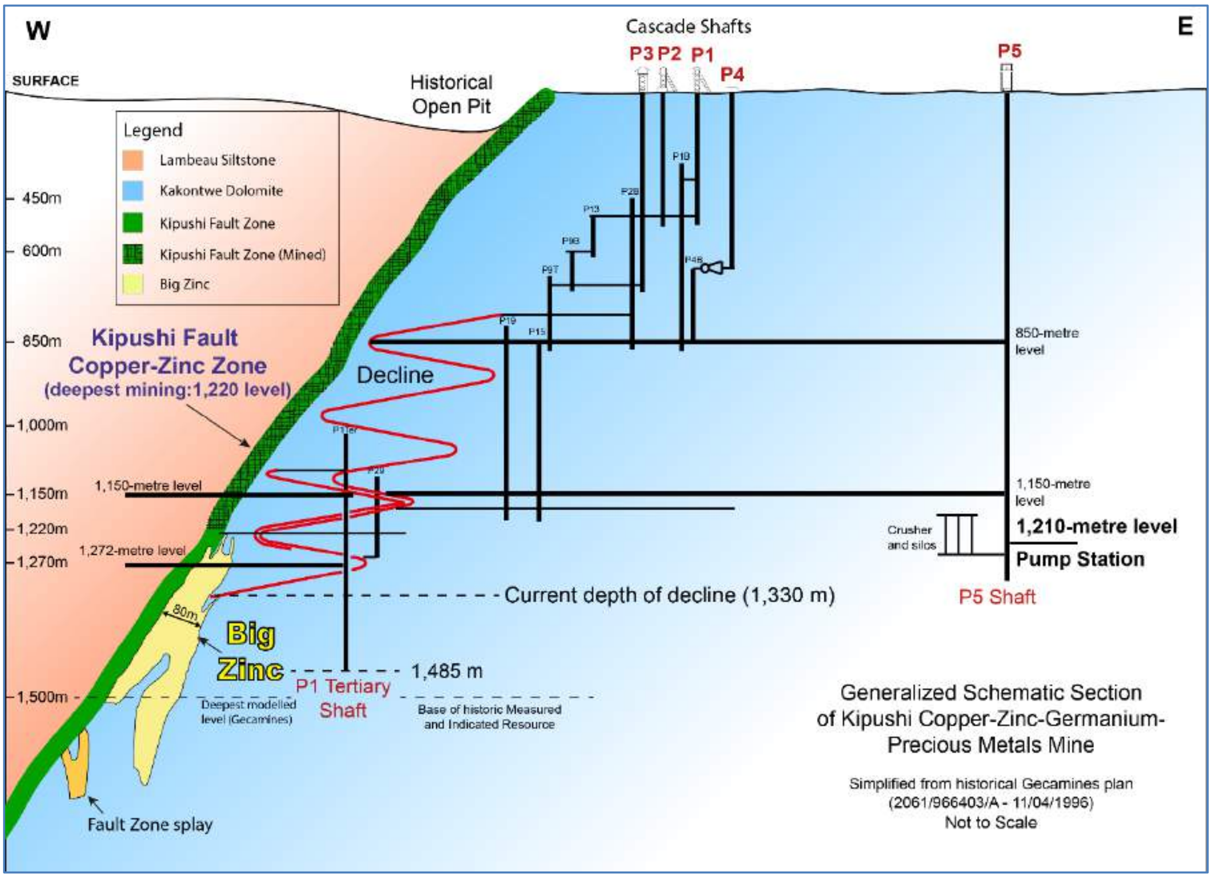


Figure by Ivanhoe, 2016.

16.2.1 Geotechnical Investigation

The geotechnical investigation and design was based on 72 geotechnical drillholes for rock mass characterisation and 16 drillholes for the structural analysis. Uniaxial compressive strength (UCS) tests were also conducted on samples from the major lithological units to gain an impression of the intact rock strength of the rock.

The primary aim of the geotechnical investigations was to carry out quality control and assurance (QAQC) on the geotechnical logging, to provide geotechnical logging recommendations, to classify and analyse the rock mass for the project area, and to provide preliminary design parameters for mining method selection and conceptual mine design.

A summary of the findings for the geotechnical investigation is as follows:

- The rock units in the Kipushi area generally have a mean intact compressive strength of greater than 200 MPa.
- The mean rock densities in the Kipushi area range from 2.75 g/cm^3 to 2.85 g/cm^3 for the host rock and overall, the host rock has a mean density of 2.79 g/cm^3.
- The mean rock densities in the Kipushi area range from 4.03 g/cm^3 to 4.20 g/cm^3 for the mineralised rock.
- Overall, the Kipushi Fault and Big Zinc mineralised zones have a higher rock mass quality than that of the Série Récurrente zone.
- Overall, the rock mass quality of the footwall is lower than that of the hangingwall and the mineralisation.
- The Série Récurrente zone footwall has the lowest rock mass quality.
- Based on the structural analysis undertaken, two joint sets have been identified.
- An underground visit confirmed that the ground conditions are very good and most of the underground excavations observed were stable without support, with negligible stress damage.

The results from the geotechnical investigation have been used to provide preliminary geotechnical design parameters, which can be used for mining method selection and conceptual mine design. The analyses included stope stability, considerations for pillars and/or backfill, assessments into the caveability of the hangingwall and the stability of mine access development.

The preliminary findings for the geotechnical design are as follows:

- Transverse stopes will be more stable than longitudinal stopes, mainly due to the major principal stress orientation in the stope back.
- If a bottom-up Sublevel Open Stoping (SLOS) method is used, a diminishing sill pillar will be formed and the stresses in longitudinal stope backs will increase as the pillar reduces in size. Extraction of the sill pillars between levels may be possible, but the production rate should be reduced to at least 50%, with higher support costs. An additional tonnage loss of at least 20% should be considered within the final crown pillar. Pillar recovery of 60% should be assumed. A cement content of 12% (approximately 2.2 MPa) over a 10 m height of the stope above the crown pillar should be assumed.
- Maximum hydraulic radii have been provided for the upper (80 percentile) and lower (20 percentile) bound rock mass conditions. The lower bound hydraulic radius for longitudinal stopes can be as low as 6 m for the backs and 4 m for the Série Récurrente zone footwall. The other lower bound hydraulic radii exceed 10 m. The upper bound hydraulic radii are invariably more than 25 m. Strike parallel joints also affect the stability of the vertical walls of longitudinal stopes in Série Récurrente zone.
- Ore drives will require support in the form of split sets and mesh to cater for stress damage. During extraction of sill pillars, it is recommended that cable anchors are also installed.

- The development infrastructure is all situated in the footwall and not subjected to stress damage. Where the decline is within 20 m of a sill pillar, mesh or shotcrete will be required to contain stress damage.
- The support analysis indicates that most of the footwall development will not require support but, for safety purposes, it is considered prudent to install split sets.

In the next stage of the project, the following work is recommended:

- All assumptions will need to be verified with more data and analysis.
- Additional UCS testing per rock unit, where failures are photographed and failure mechanisms are investigated further.
- A full suite of laboratory testing, including triaxial compressive strength tests and Brazilian disc tests are also recommended to investigate the rock properties of all rock units.
- Further geotechnical drilling and logging will be required in the next stage of the project to increase the confidence in geotechnical data.
- The direction of drilling in the next stage should be along strike to avoid an orientation bias, as the majority of drilling at this stage is in the dip direction of the various mineralised zones.
- Underground mapping should be carried out to improve confidence in the joint orientations and rock mass classification.

16.3 Mine Design

The Big Zinc zone is located at depths ranging from approximately 1,185–1,710 mRL. Access is expected to be via the existing vertical shafts and the internal decline. The existing decline is planned to be extended from the current position. Development and stope production is expected to be hauled by loaders to the decline and loaded into trucks. From the levels the trucks are expected to haul material to the 1,150 mRL drive. Rail haulage is planned for the 1,150 mRL main haulage level to the crusher at Shaft 5 for hoisting to the surface. The assumptions for Kipushi development are shown in Table 16.1.

Table 16.1 Kipushi Mine Development Assumptions

Description	Height (m)	Width (m)	Comment
Decline	5.0	5.8	Gradient 1-in-7; Radius 35 m
Ore Drives	5.0	5.0	–
Access	5.0	5.0	–
Fresh Airways	5.0	5.0	–
Waste Pass Access	5.0	5.0	–
Vertical Development	–	–	Longhole Raise 4 m diameter
Development Stockpile	5.0	5.8	Length 15 m every 80 m

Kipushi Big Zinc zone and Copper zone stopes, existing and planned development and shafts are shown in Figure 16.2 and Figure 16.3.

Figure 16.2 Planned and Existing Development at Kipushi

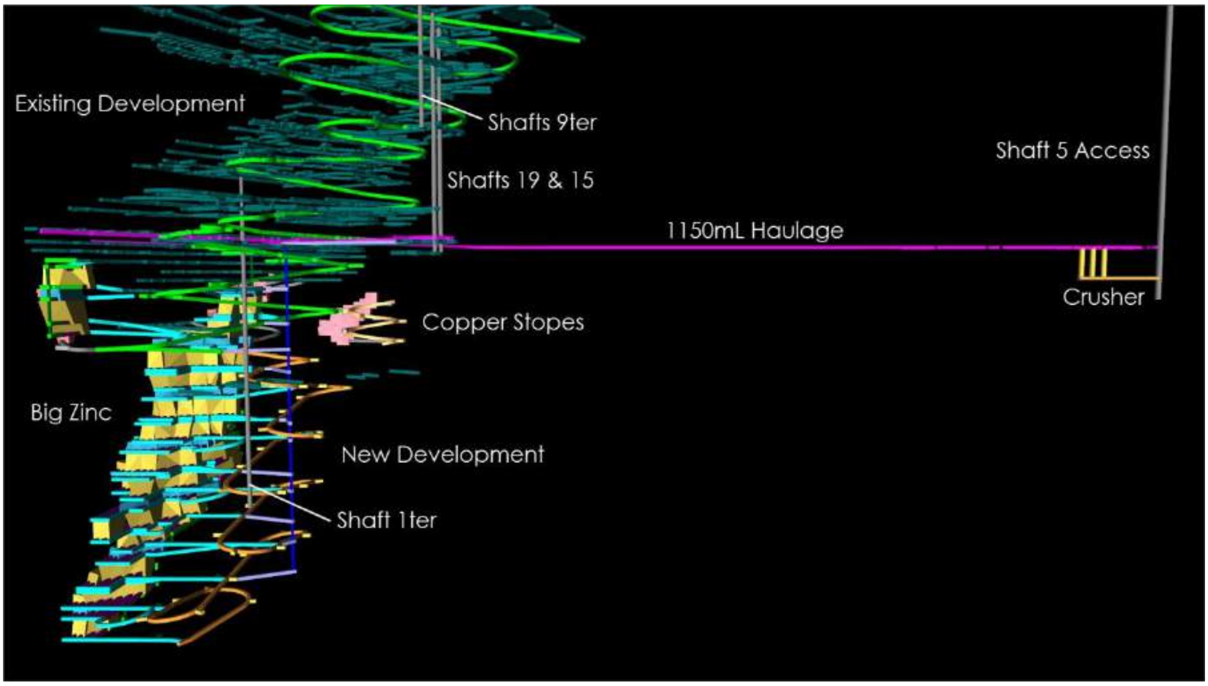


Figure by OreWin, 2016.

Figure 16.3 Planned and Existing Development at Kipushi

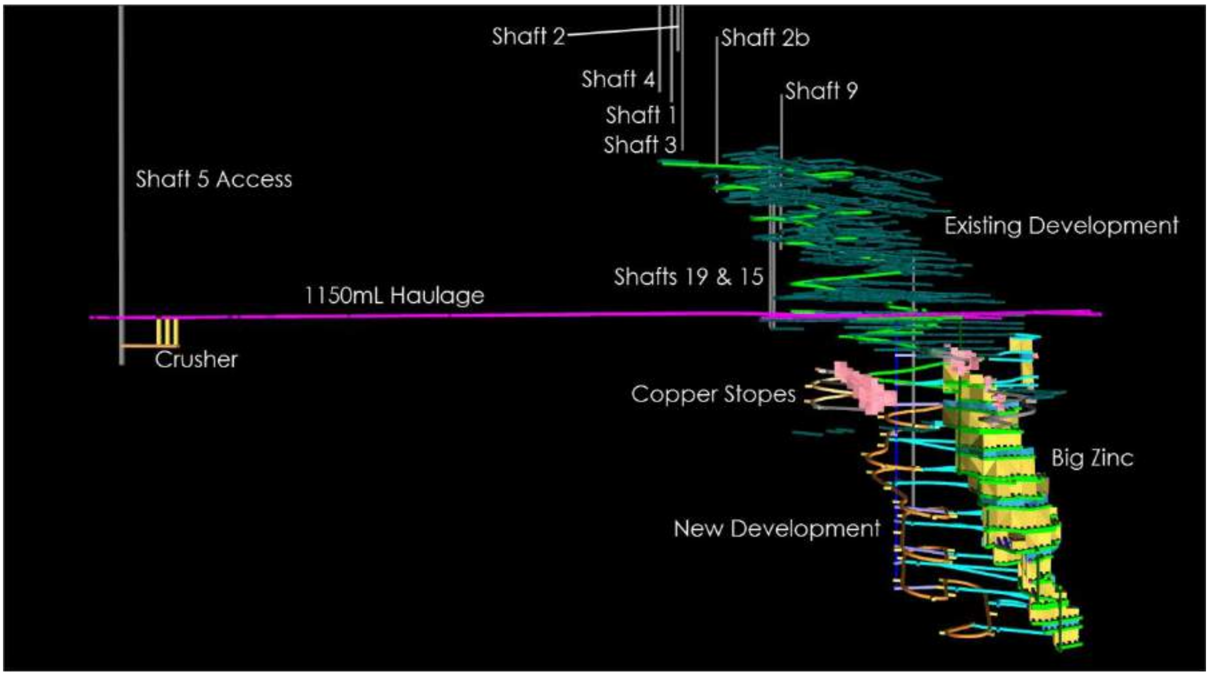


Figure by OreWin, 2016.

16.4 Mining Methods

Mining is planned to be a combination of longitudinal or transverse Sublevel Open Stoping (SLOS), Pillar Retreat, and Cut and Fill methods. The Big Zinc zone mining method is expected to be longitudinal SLOS with mined stopes backfilled with cemented rock fill after stoping. The crown pillars are expected to be mined using the Pillar Retreat mining method once the adjacent stopes are backfilled. The Copper zone outside of the Big Zinc zone is expected to be mined by the Cut and Fill mining method.

The Big Zinc zone 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 mRL to the bottom stoping level at 1,845 mRL. The zinc stoping is expected to be carried out between 1,215 mRL and 1,845 mRL, and the uppermost stoping level on the Big Zinc zone is planned to be the 1,290 mRL. 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 stope is planned to be drilled via two parallel drives in each stope. The crown pillar height is planned to be 15 m. Stopes are planned to be mined 20 m along strike and then filled with a cement rock fill. Remote capable loaders are expected to be used for loading the broken rock beyond the stope brow.

There is a small amount of copper tonnes (168 kt) planned to be produced by SLOS from the resource that is within the Big Zinc zone. These stopes are expected to be mined and filled separately to the main stoping producing zinc tonnes. The assumptions for SLOS are shown in Table 16.2. The average production rate for a single SLOS stope heading was estimated to be 224 ktpa. This includes the cycle of activities: drilling, blasting, mucking, and backfilling. On the widest stoping levels, it could be possible to produce from up to three stopes on a level and have two levels in operation. The Pillar Retreat stoping does not require backfill and has been estimated at a production rate of 504 ktpa. This rate is slower than the SLOS rate when backfill (including curing time) is excluded. Figure 16.4 and Figure 16.5 respectively show transverse and longitudinal cross-sections of the SLOS method.

Table 16.2 Kipushi Sublevel Open Stoping Parameters

Parameter	Unit	Amount
Longitudinal Sublevel Open Stoping		
Production Rate per Stope	ktpa	224
Main Level Interval	m	60
Sublevel Interval	m	30
Stope Width	m	20
Drive Height	m	5
Drive Width	m	5
Drive in each stope	Ea.	2
Stope Dilution	m	2
Mining Recovery	%	90
Backfill Dilution	%	2.5
Pillar Retreat		
Production Rate per Stope	ktpa	504
Crown Pillar Interval	m	15
Pillar Retreat Recovery	%	60
Pillar Retreat Dilution	%	20

Figure 16.4 SLOS Mining Method – Transverse Cross-Section

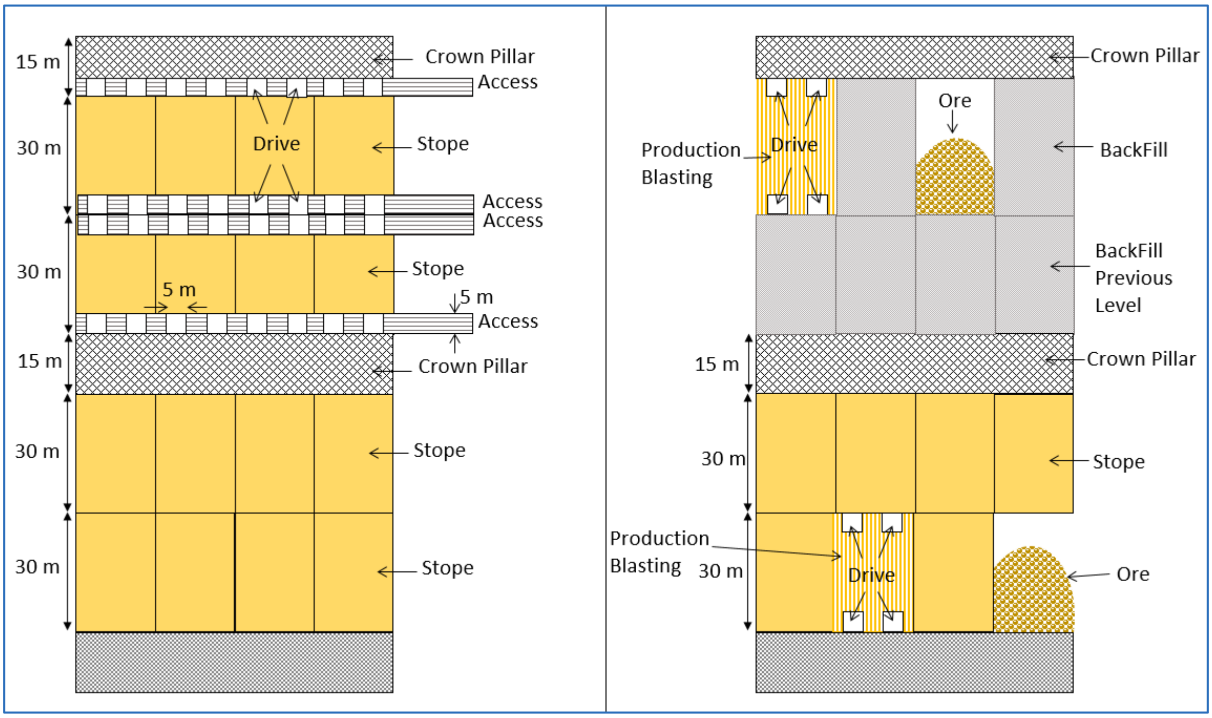


Figure by OreWin, 2016.

Figure 16.5 SLOS Mining Method – Longitudinal Cross-Section

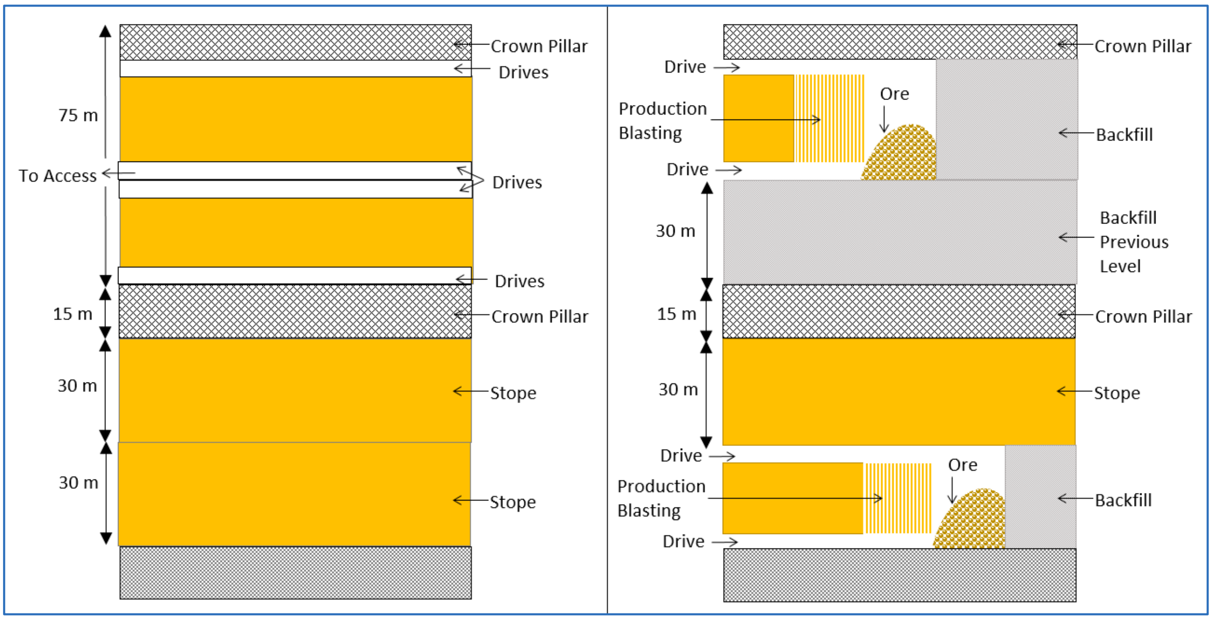


Figure by OreWin, 2016.

Figure 16.6 and Figure 16.7 show the Kipushi zinc stope and development plans at 1,440 mRL and 1,485 mRL respectively.

Figure 16.6 Kipushi Longitudinal Stope and Development Plan at 1,440 mRL

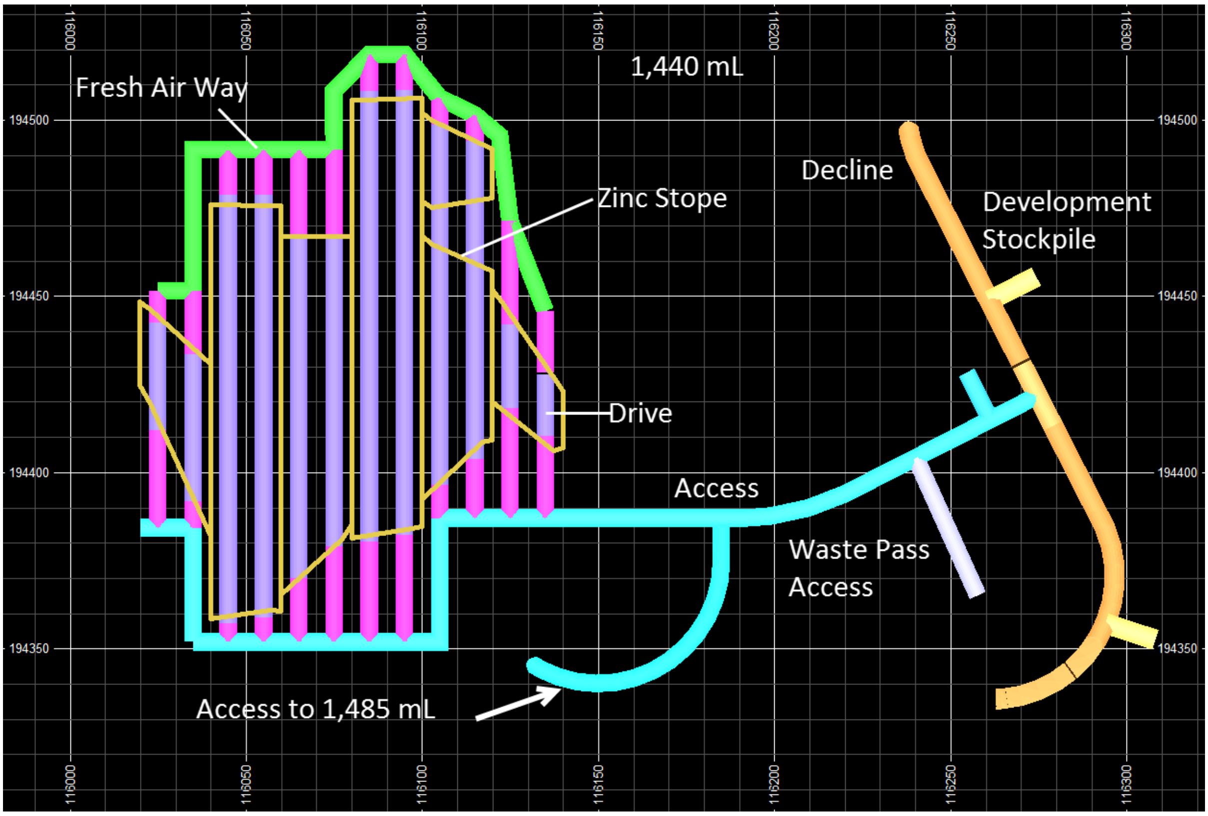


Figure by OreWin, 2016.

Figure 16.7 Kipushi Longitudinal Stope and Development Plan at 1,485 mRL

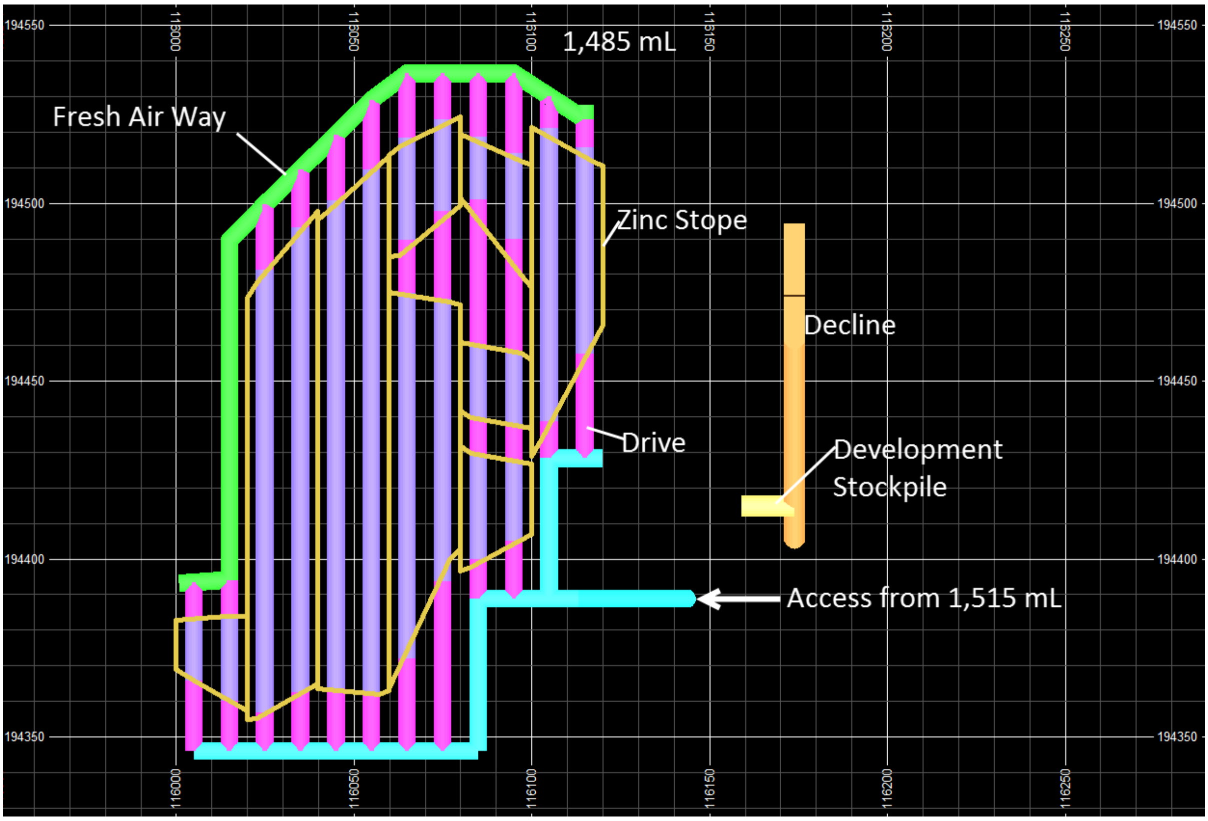


Figure by OreWin, 2016.

A Cut and Fill mining method has been identified to be used in the Copper zone stopes planned from 1,210 mRL to 1,330 mRL. In this method, mining occurs in horizontal slices with the blasted material loaded and removed from the stopes. When the stope is mined out the resultant stope void will be backfilled to allow the next horizontal slice to be mined above. The parameters of the Cut and Fill mining method are shown in Table 16.3. A schematic section of the Cut and Fill mining method is shown in Figure 16.8.

Table 16.3 Kipushi Drift and Fill Parameters

Parameter	Unit	Amount/Type
Production Rate	ktpa	168
Stope Access	–	Footwall ramp
Level Interval	m	20.0
Maximum Ramp Inclination	–	1 in 5
Slice Height	m	5.0
Face Advance	m	4.0

Figure 16.8 Cut and Fill Mining Method Schematic Section

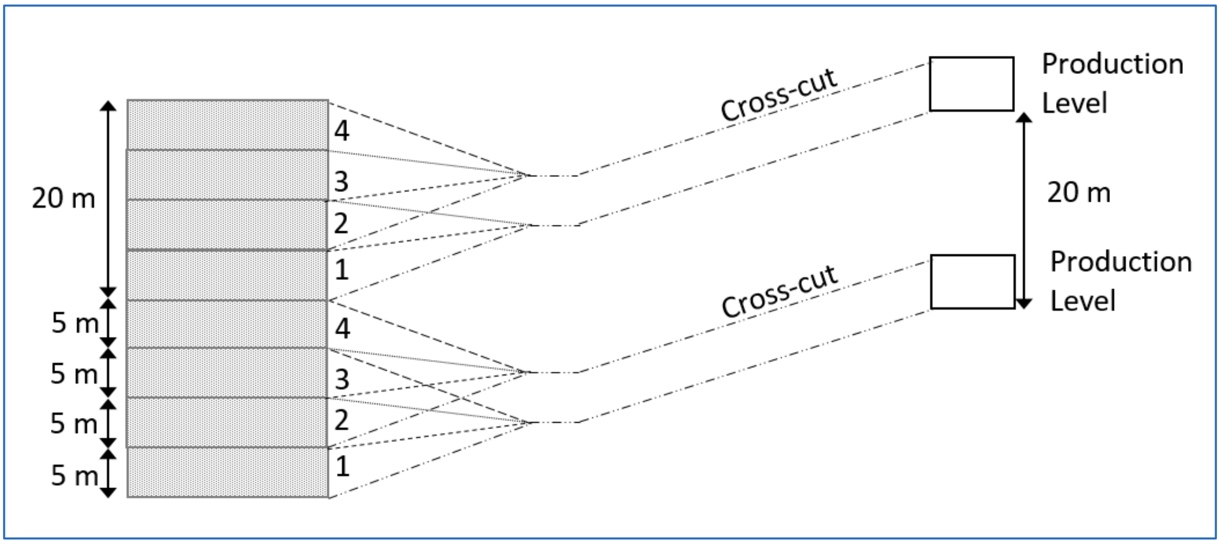


Figure by OreWin, 2016.

Backfill is planned to be sourced from the plant tailings. The slimes tailings are expected to be mixed with the coarse DMS tailings. The slimes material represents only 2% of the tailings; therefore, it has been assumed that it will have no significant impact on the backfill flow through the passes. Tailings are planned to be trucked from the plant directly to Shaft 3 or to the tailings stockpile area. Shaft 3 is planned to be stripped and extended to be used as a waste pass. From the 850 mRL waste is planned to be trucked to 1,138 mRL and tipped into the P1 Tertiary shaft, which could be used as an internal backfill pass allowing transfer of waste to the stope voids.

Cement is expected to be added to the rock fill before it is delivered by loader into the mined out stopes. On each level two mixing sumps are planned to be developed and one loader is expected to deliver dry fill into the sumps. A diagram of the mixing sump layout is shown in Figure 16.9. Agitator trucks are expected to transfer the cement batch from a slurry plant on 1,330 mRL. A second loader is expected to tram the fill from the mixing sump. Remote capable dozers are expected to be used to push the cement rock fill inside the open stopes.

Figure 16.9 Backfill Mixing Sump Layout

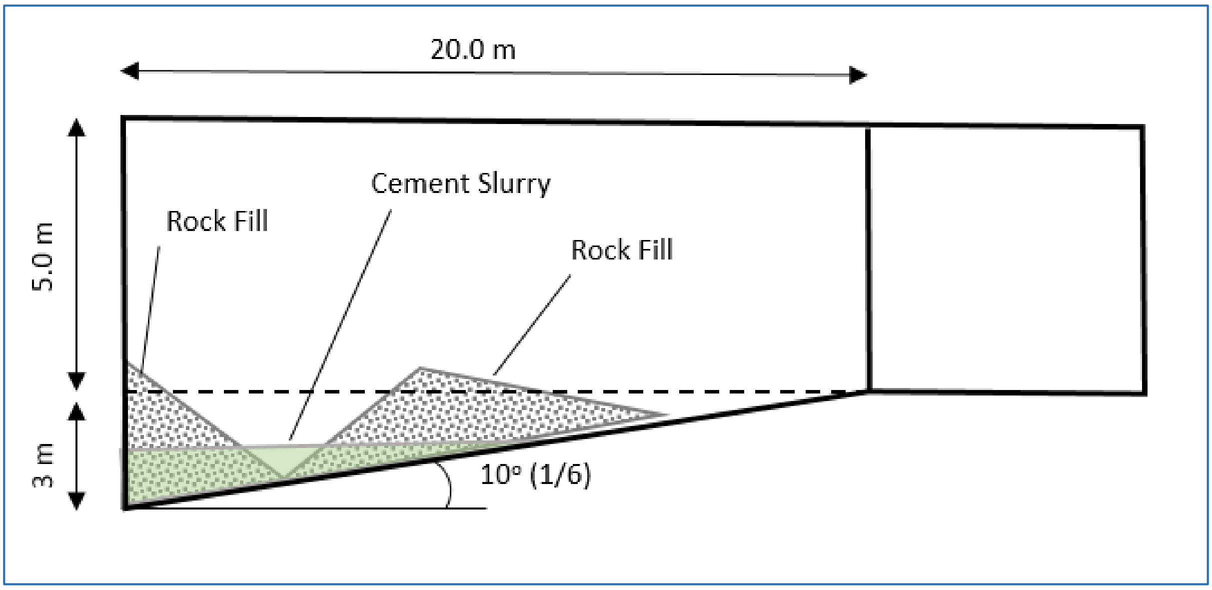


Figure by OreWin, 2016.

The key underground mine equipment is:

- Development Drill
- 17 t Diesel LHD
- 51 t Dump Truck
- Support Bolting
- Bulldozer
- Production Drill
- Scissor Lift
- Underground Grader
- Explosive Cassette Carrier
- Explosive Charger
- 4WD LDV - Explosives
- 4WD LDV
- Passenger Transporter
- Shaft Maintenance
- Lube / Fuel Truck
- Pallet Handler
- Skip Bin Loader
- Tipper Truck

- Wheel Handler
- Wheel Loader
- Agitator Truck
- Slurry Mixing Plant
- Rail Haulage Locos
- Rail Cars
- 760/45 kW Fan
- Submersible Pumps 400 kW
- Submersible Pumps
- Skid Tank Vertical Spindle Pumps

The planned numbers of development jumbos, loaders, production drills, and trucks are shown in Figure 16.10 to Figure 16.13. Fixed equipment allowances have also been included in the mine plans.

Figure 16.10 Mobile Equipment In-Service – Development Drills

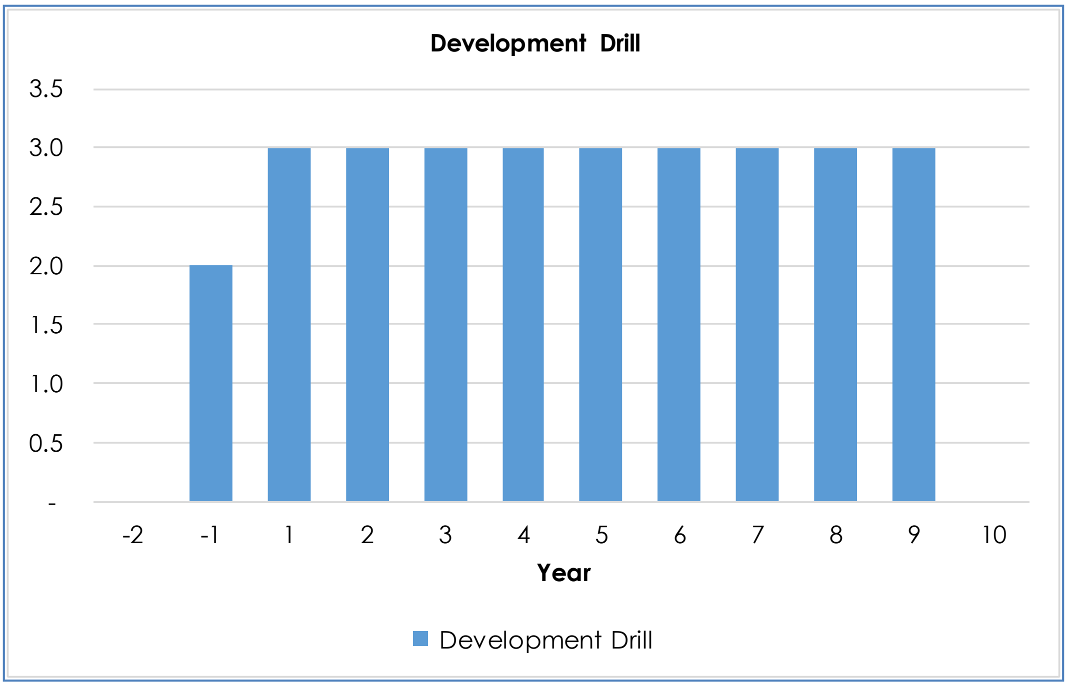


Figure 16.11 Mobile Equipment In-Service – Loaders

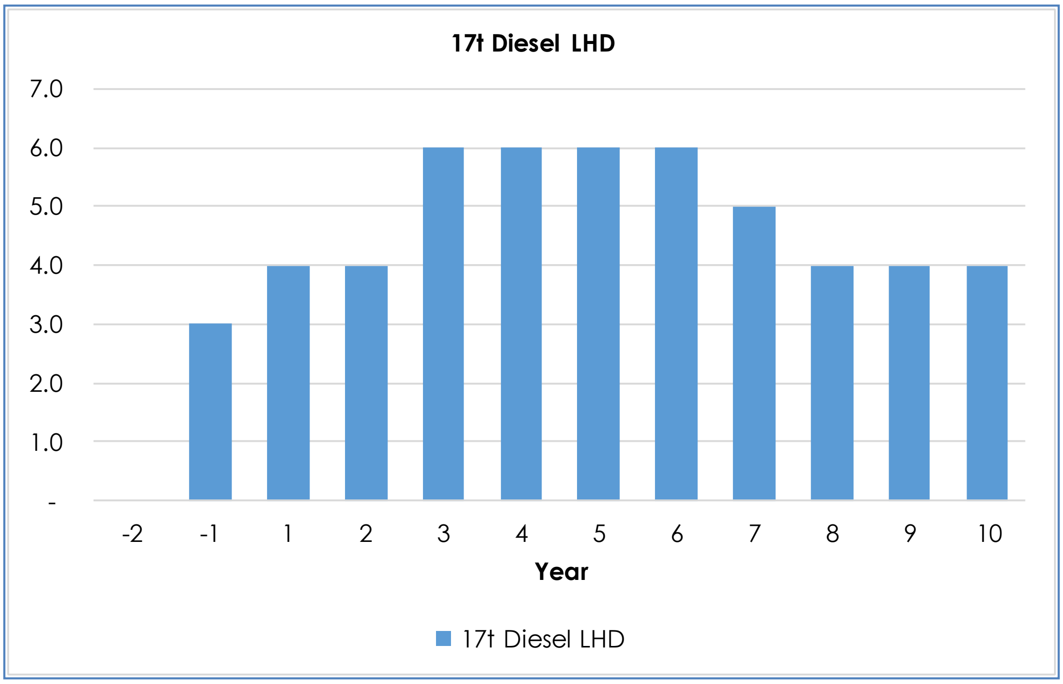


Figure 16.12 Mobile Equipment In-Service – Production Drills

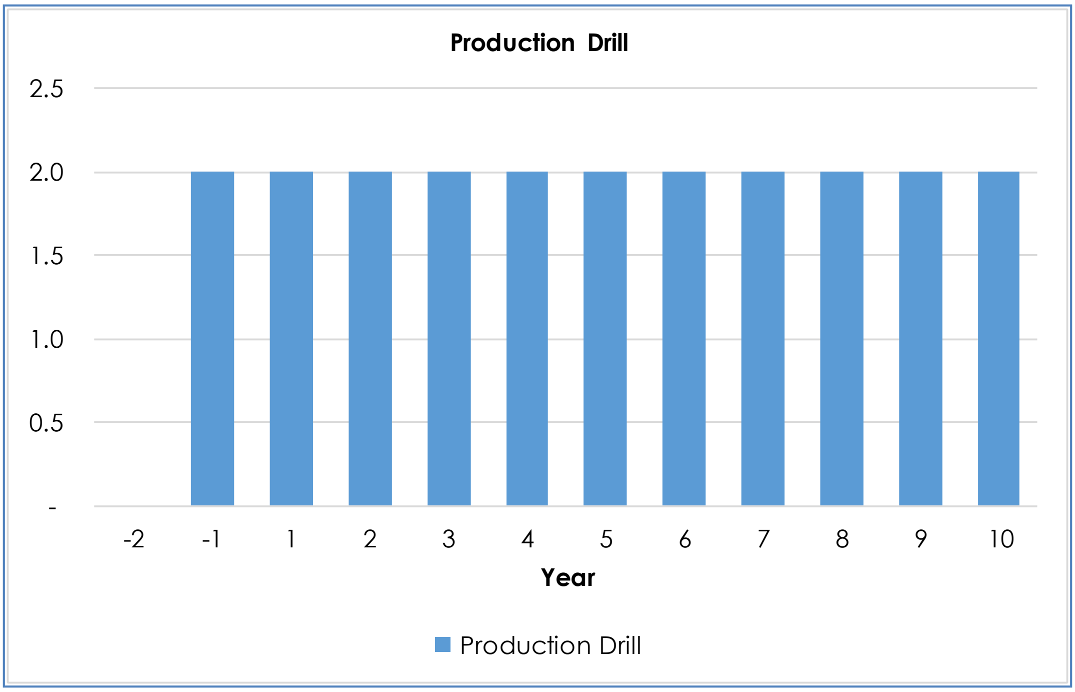
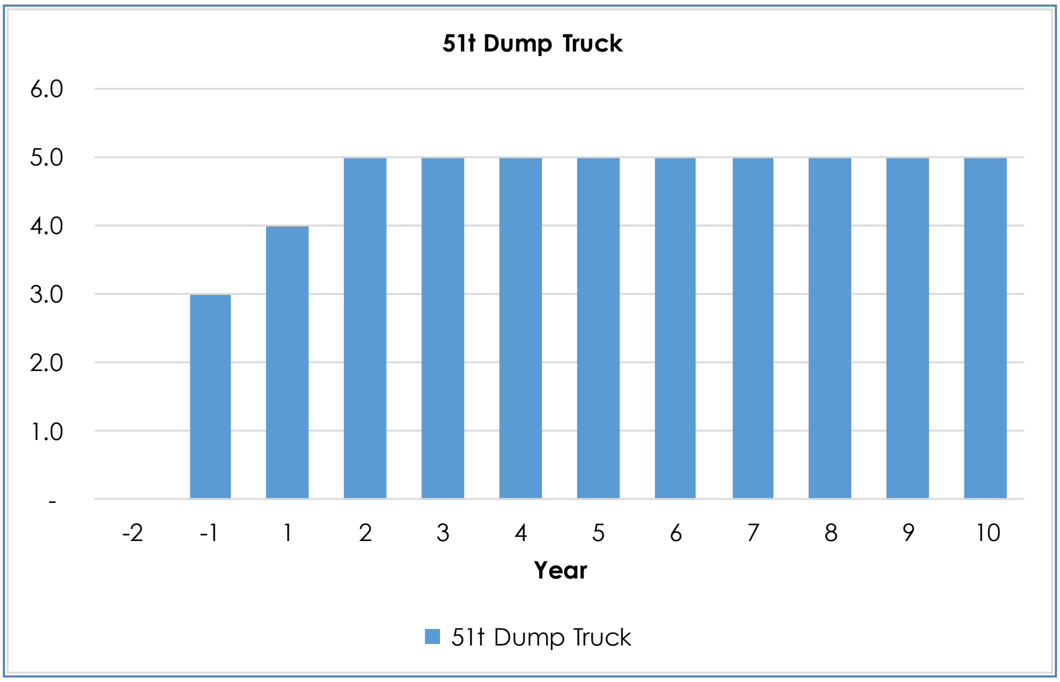


Figure 16.13 Mobile Equipment In-Service – Trucks



The planned Kipushi development and production schedules are summarised in Table 16.4 and Table 16.5.

Table 16.4 Kipushi Development Schedule Summary

	Units	Total	Schedule Year									
			–1	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**
Lateral Development												
Decline	m	**5,849**	959	959	765	426	959	958	–	822	–	–
Access	m	**9,326**	1,102	772	1,324	1,297	919	448	1,446	1,353	579	86
Ore Drive	m	**15,272**	830	1,732	1,667	1,938	1,522	2,101	1,827	1,645	1,865	145
Waste Drive	m	**13,785**	656	1,138	1,268	1,515	1,444	1,824	1,845	1,193	2,592	308
Fresh Air Way	m	**6,378**	670	475	669	648	812	420	767	659	700	558
Waste Pass Access	m	**893**	365	97	114	106	115	96	–	–	–	–
Total	**m**	**51,501**	**4,582**	**5,173**	**5,807**	**5,930**	**5,771**	**5,846**	**5,885**	**5,673**	**5,737**	**1,097**
Vertical Development												
Fresh Air Raise	m	**768**	210	25	5	75	59	91	108	45	135	15
Waste Pass	m	**480**	185	100	–	50	100	45	–	–	–	–
Total	**m**	**1,248**	**395**	**125**	**5**	**125**	**159**	**136**	**108**	**45**	**135**	**15**

Table 16.5 Kipushi Production Schedule Summary

	Units	Total	Schedule Year										
			–1	1	2	3	4	5	6	7	8	9	10
Zinc Tonnes Mined	**kt**	**9,394**	**71**	**740**	**804**	**1,089**	**1,100**	**1,099**	**1,099**	**1,098**	**1,099**	**1,100**	**95**
NSR DMS	$/t	291.3	277.2	265.1	297.5	277.9	313.3	310.6	293.8	291.3	284.6	284.5	253.3
NSR Cu	$/t	20.7	33.3	27.7	24.0	30.6	21.7	20.6	21.3	13.9	17.1	12.9	8.7
Zn grade	%Zn	32.1	31.1	29.8	33.1	30.9	34.4	34.2	32.4	32.0	31.2	31.1	27.9
Cu grade	%Cu	0.5	0.9	0.7	0.6	0.8	0.6	0.5	0.6	0.4	0.5	0.3	0.2
Copper Tonnes Mined	**kt**	**547**	**–**	**17**	**166**	**171**	**50**	**36**	**22**	**14**	**72**	**–**	**–**
NSR DMS	$/t	64.4	–	51.9	68.6	34.5	60.4	37.3	104.9	102.4	125.1	–	–
NSR Cu	$/t	203.8	–	294.7	252.4	189.1	169.3	178.5	177.8	153.7	159.1	–	–
Zn grade	%Zn	8.0	–	6.5	8.6	4.6	8.0	4.6	13.2	12.2	14.6	–	–
Cu grade	%Cu	5.4	–	7.8	6.7	5.0	4.6	4.7	4.7	4.1	4.2	–	–

17 RECOVERY METHODS

17.1 DMS Concentrator

The planned process plant is a dense media separation (DMS) plant, which is expected to include crushing, screening, HLS, and spirals to produce a high grade zinc concentrate. DMS is a simple density concentration technique that preliminary testwork has shown yields positive results for the Kipushi material, which has a sufficient density differential between the gangue (predominantly dolomite) and mineralisation (sphalerite). Figure 17.1 shows the overall proposed process plant flowsheet (block flow diagram).

Figure 17.1 Overall Proposed Plant Flowsheet

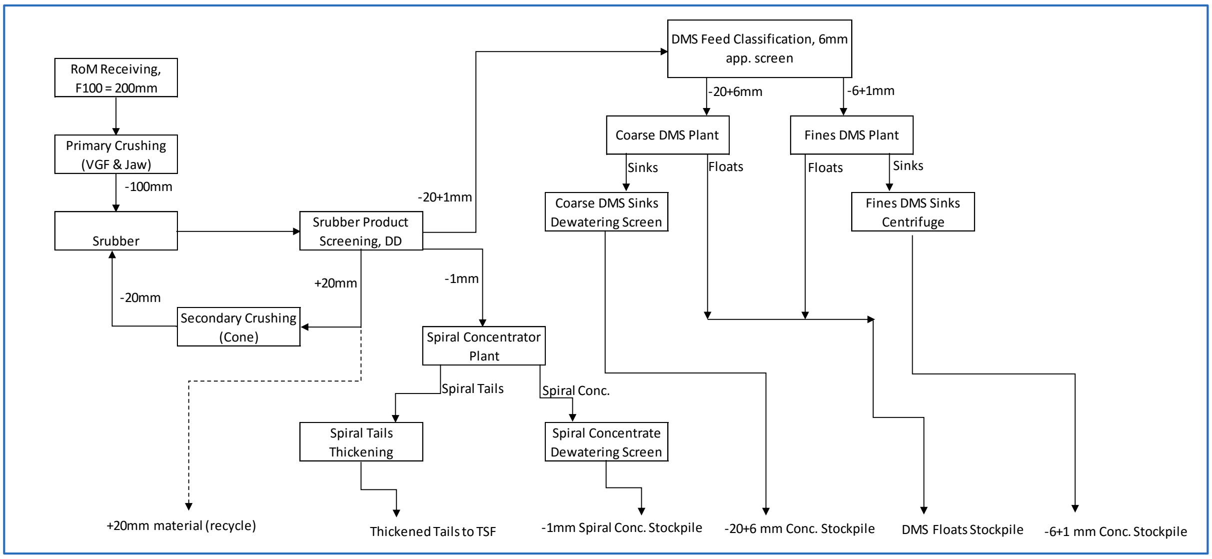


Figure by OreWin, 2016.

17.1.1 Primary Screening and Crushing Section

The crushed mined tonnes are expected to be conveyed from Shaft 5 and stockpiled on the ROM pad and fed to the plant with a Front End Loader (FEL). The FEL is expected to tip the material onto a static inclined Grizzly with 250 mm spacing between the grizzly bars.

The Grizzly oversize would fall to a stockpile which is expected to be periodically loaded by the FEL and transported back to the ROM stockpile where the large particles could be manually broken and later fed into the plant again.

The static grizzly undersize is expected to pass onto the Vibrating Grizzly Feeder (VGF) with bar spacing of 100 mm. The VGF undersize would fall onto Conveyor Belt (CV1) feeding the Scrubber and the VGF oversize would discharge into the Primary Jaw Crusher with a Closed Side Setting (CSS) of 80 mm.

The discharge from the Primary Jaw Crusher is expected to fall onto and combine with the –100 mm material on the Conveyor Belt (CV1) feeding the Scrubber. Conveyor Belt (CV1) is planned to have a Load Cell type Belt Weightometer to record tonnage.

17.1.2 Scrubber and Secondary Crusher Section

Conveyor Belt (CV1) is planned to discharge the –100 mm material into the Chute feeding into the Scrubber. A belt scraper on the head pulley of the conveyor belt could eliminate spillage on the conveyor belt return side. Process water is expected to flush the -100 mm material from the Chute into the Scrubber. The process water valve is planned to be manually adjusted to control the slurry viscosity and density in the scrubber. Spray water from the DMS Plants is also planned to enter the Scrubber feed. In the Scrubber the clay particles are expected to be reduced in size due to the scrubbing action of the particles and rocks against each other whilst the scrubber drum is rotating. The Scrubber is expected to discharge the slurry onto a double deck Vibrating Screen (VS1) with 20 mm aperture polyurethane screen panels on the top deck and 1 mm aperture polyurethane screen panels on the bottom deck. High pressure Spray Water Nozzles above each deck are planned to wash the fine particles off the larger particles retained on the screen decks. The screen spray water valves are planned to be manually adjusted.

The vibrating screen +20 mm particles are planned to discharge off the top deck and pass down a chute and onto a Conveyor Belt (CV2) which would discharge into a 30 t Surge Bin. A Feeder is planned to draw crushed material out of the Surge Bin and discharge onto a Conveyor Belt (CV3), which would discharge into the secondary cone crusher with a CSS of 20 mm. The discharge from the Secondary Cone Crusher is planned to fall onto a Conveyor Belt (CV4) then Conveyor Belt (CV5) feeding the Scrubber feed chute. There is an allowance for excess +20 mm material from VS1 to discharge into a slewing conveyor (CV16), the material could be reclaimed by FEL and recycled back to the scrubber.

The vibrating screen –20+6 mm particles, discharged off the bottom deck, are planned to pass down a chute and onto a Conveyor Belt (CV6) which is planned to discharge onto a single deck Vibrating Screen (VS2) with 6 mm polyurethane screen panels. The screen oversize particles are planned to discharge into a 60 t, –6+1 mm Surge Bin.

It is expected that all the screen washings (–1 mm particles and water) gravitate into the screen under pan. A 6 inch by 4-inch Centrifugal Pump connected to the under pan is planned to pump the slurry to a Settling Cone with another 6 inch by 4-inch Centrifugal Pump planned to pump the settled slurry to a 20 tph Spiral Concentrator Plant.

It is expected that a sump pump will be used to return the Scrubber section floor spillage to the top deck of the double deck vibrating screen.

17.1.3 Spiral Concentrate Plant

A spiral concentrator plant is planned to upgrade the zinc sulphide in the –1 mm size fraction through three stages of upgrading. A dewatering screen is expected to remove most of the water from the zinc concentrate. The zinc concentrate from the dewatering screen oversize is planned to discharge onto a conveyor belt (CV14), then onto a slewing stacking conveyor (CV15), which is planned to discharge onto the spiral product stockpile. This material is expected to be discharged into a concrete pad and allowed time to self-drain before being bagged. A load cell type weightometer is planned to be used on the conveyor belt (CV14) to record the spiral plant zinc production. The tailings from the spiral plant together with the water from the dewatering screen is expected to be fed to the tailings thickener for water recovery. Thickener overflow is planned to gravitate into a process water tank and thickener underflow is planned to be pumped to the Slimes Dam.

17.1.4 DMS Plant Section

Two 75 tph cyclone type DMS plants are planned. The two DMS plants are expected to be placed parallel to each other with DMS Plant 1 being fed the –20+6 mm particles and DMS Plant 2 being fed the –6+1 mm particles.

The feed to each DMS Plant is planned to be drawn out of the respective DMS Feed Bins by Feeders that discharge onto Conveyor Belts (CV7 and CV8) that discharges onto the Feed Preparation Screen (Prep Screen) of the respective DMS Plants. The Feeder feed rate to the DMS Plant is planned to be mechanically adjusted. High pressure Spray Water is expected to wash any misplaced –1 mm fines off the particles on the Prep Screen and these screen undersize washings are planned to be pumped back to the Scrubber feed chute. The Prep Screen oversize is planned to discharge into the Mixing box. Ferro silicon (FeSi) from the FeSi Media Tank is planned to be pumped into the Mixing Box. The feed and FeSi mixture is then planned to be pumped from the Mixing Box to the DMS Cyclone. The level of the feed and FeSi mixture in the Mixing Box is planned to be manually controlled by adjusting the FeSi make-up valve. In the DMS Cyclone the zinc sulphide mineral is expected to be separated from the gangue due to the different specific gravities and the relative density of the FeSi media. The relative density of FeSi media is selected through laboratory testwork and actual plant testwork to produce the desired grade of the zinc product. The DMS cyclone feed pressure is planned to be measured and monitored.

The DMS Cyclone underflow is planned to flow to the Sinks Rinse / Drain Screen with 1 mm aperture wedge wire screen panels. High pressure water Spray Nozzles are expected to wash FeSi media off the particles. The Sinks screen oversize from the –20+6 mm DMS Plant is planned to discharge through a chute onto a Conveyor Belt (CV9) which feed into the dewatering screen to reduce moisture content of the coarse DMS product. The dewatering screen oversize is planned to discharge with expected moisture content approximately 8% through a chute onto a conveyor belt (CV10). A load cell type weightometer is planned for the Conveyor Belt (CV10) to record the zinc production from the –20+6 mm DMS Plant. The sinks screen oversize from the –6+1 mm DMS Plant is planned to discharge through a chute onto a Conveyor Belt (CV11) which is planned to feed into the centrifuge to reduce moisture content locked on the fine material. The centrifuge is expected to discharge relatively dry product with the moisture content <6% through a chute onto a conveyor belt (CV12). A load cell type weightometer is planned for Conveyor Belt (CV12) to record the zinc production from the -6+1 mm DMS Plant.

The DMS Cyclone overflow is planned to flow to the Floats Rinse / Drain Screen with 1 mm aperture wedge wire screen panels. High pressure water Spray Nozzles are expected to wash FeSi media off the particles. The Floats screen oversize from each DMS Plant is planned to discharge through chutes onto a Conveyor Belt (CV13) which is planned to discharge onto the combined Floats / Discard stockpile.

The undersize from both the Sinks and the Floats rinse/drain screens is planned to flow through a chute to the Magnetic Separation (Mag Sep) Drum where the FeSi media is expected to be recovered from the water and returned to the Media Tank. The water is planned to be combined with the fines from the Prep Screen which is planned to be pumped to the Scrubber feed chute. The scraper on the Mag Sep Drum is expected to be checked and adjusted periodically.

The FeSi Media Tank is planned to be mechanically stirred to prevent the media from settling out and solidifying in the tank. The density of the FeSi media in the Media Tank is planned to be controlled by pumping the FeSi media through an in-line nuclear Densitometer to measure the media relative density and into a Tube Densifier. The centrifugal force in the Tube Densifier separates the water from the FeSi media and Tube Densifier underflow with dense media is returned to the Media Tank whilst the Tube Densifier overflow with the minor amounts of FeSi reports to the Mag Sep Drum. A signal from the Densitometer is planned to control an auto water valve to add water to the Media Tank to maintain the desired FeSi media relative density.

It is expected that a sump pump will be used to return the DMS section floor spillage to the top deck of the Scrubber vibrating screen.

17.1.5 FeSi Make-up

During operations FeSi media could sometimes be lost and fresh FeSi media needs to be added to the DMS plant. The FeSi media is planned to be made-up by adding new FeSi powder to the DMS Plant Media Tank.

17.1.6 Flocculent Make-up

The 15 m Thickener is planned to use flocculent to assist with solids settling and production of clear water for return to the process. The flocculent make up is planned to consist of a simple bag splitter for 25 kg bags and mixed with water to achieve the desired flocculent mix. The make-up is planned to include storage and metering pumps.

17.1.7 Process Water

It is planned for a process Water Storage Tank and Pump to supply water to the Scrubber section and the DMS Plant Section. The process water needs to be free of suspended solids for efficient operation of the plant. Hosing points will enable any spillage to be washed down.

17.1.8 Electrical and Instrumentation

Electricity is planned to be supplied from the national grid; a diesel generator is expected to provide back-up 525 Volt electrical power to allow controlled stoppages in the event of power interruptions. The diesel generator is planned to only allow an orderly shutdown of the DMS plant, it is not planned to operate the whole process plant in the advent of a power outage.

A containerised Motor Control Centre (MCC) with an adjacent transformer houses all the electrical switchgear components is planned.

Instrumentation signals are expected to be digital data on a single wire to and from the Programmable Logic Controller (PLC).

17.1.9 Operator Cabin Control

It is planned that a containerised operator control cabin mounted above the MCC will provide a commanding view of the plant and house the PLC and the Supervisory Control and Data Acquisition (SCADA) unit to control the operation.

17.1.10 Process Tailings

There are two tailings streams in the planned process plant; coarse DMS tailings and fine Spiral tailings. It is expected that all tailings will be utilised as underground backfill.

The coarse DMS tailings, nominally –20+1 mm, material is expected to be picked up by FEL and placed into trucks. The trucks are planned to transport the material to either; Shaft 3 where it will be sent underground, or the coarse tailings stockpile area if there is insufficient capacity at Shaft 3.

The fine Spiral tailings material, nominally –1 mm, after thickening are planned to be pumped to one of two settling dams. The settling dams, one in use and one being reclaimed, are expected to be a temporary storage and dewatering location for the tailings. Reclaimed water is planned to be returned to the process plant. The settled fine tailings are expected to be reclaimed and combined with coarse DMS tailings and used as underground backfill.

17.2 Concentrate Handling

Zinc concentrate product from the DMS and Spiral concentrator are expected to be combined, using front-end loaders and conveyed to a planned semi-automated concentrate bagging facility adjacent to Shaft 5. The concentrate after centrifuging and dewatering is expected to contain nominal 8% moisture.

The planned bagging plant, consisting of 2 x 60 tph streams will bag into 2 t bulk bags. It is expected that samples of every second bag will be taken for analysis. Bulk bags of nominal 900 x 900 mm footprint and nominally 850 mm height are expected to contain 2 t at a nominal density of 3.0 g/cm^3.

It is planned that bags will be numbered and labelled and loaded directly onto trucks, nominally 20 t capacity, to be transported to the planned container loading area adjacent to Kipushi Station. The nominal distance is 2 km from concentrate bagging at Shaft 5 to the container yard adjacent to the Kipushi Station.

Trucks are expected to be loaded at a rate of nominally four per hour, 24 hours per day, or 98 trucks per day.

17.2.1 Rail Loading

The container yard, planned to be constructed and operated by the rail contractor, is adjacent to Kipushi Station. The container yard is expected to be used to store containers and as a loading and unloading point. This yard is intended to be a bonded yard with respect to DRC Customs and import and export of good for the mine.

The Kipushi mine has an existing bonded yard, therefore it is not anticipated that either gaining approval for a second bonded yard or transferring the yard location will present an issue. In the bonded yard DRC customs, based on site, with the rail contractor's personnel are expected to undertake the necessary documentation and procedures to prepare the containers for export.

The rail contractor utilising the container yard and a dedicated hardstand area adjacent to the Kipushi Station rail network is expected to load and unload trains using container handling forklifts.

17.3 Processing Schedule

Total planned zinc plant feed from the Kipushi Underground Mine is 9.4 Mt at 32.15% Zn and is planned to be treated for ten years. Total zinc concentrate produced is 5,296 kt (dry) at 53% Zn. The planned DMS processing schedule is summarised in Table 17.1.

Table 17.1 DMS Processing Schedule

	Unit	Total	Schedule Year									
			1	2	3	4	5	6	7	8	9	10
Zinc Feed Tonnes Processed												
DMS Plant Feed	**kt**	**9,394**	**812**	**804**	**1,089**	**1,100**	**1,099**	**1,099**	**1,098**	**1,099**	**1,100**	**95**
NSR DMS	$/t	**291.29**	266.20	297.55	277.93	313.34	310.64	293.79	291.34	284.60	284.48	253.30
NSR Cu	$/t	**20.70**	28.19	23.98	30.62	21.65	20.64	21.28	13.87	17.10	12.94	8.74
Zn grade	%	**32.15**	29.89	33.09	30.92	34.43	34.19	32.42	32.04	31.24	31.13	27.94
Cu grade	%	**0.55**	0.75	0.64	0.81	0.57	0.55	0.56	0.37	0.45	0.34	0.23
Zinc Concentrate												
Zn Recovery	%	**92.94**	92.01	93.29	92.43	93.81	93.72	93.03	92.88	92.56	92.52	91.18
Concentrate Produced	kt (dry)	**5,296**	421	468	587	670	664	625	616	600	598	45
Zn grade	%	**53.00**	53.00	53.00	53.00	53.00	53.00	53.00	53.00	53.00	53.00	53.00
Zn Produced	kt	**2,807**	223	248	311	355	352	331	327	318	317	24

Planned DMS plant feed, zinc concentrate and quantity of metal produced are shown in Figure 17.2 and Figure 17.3.

Figure 17.2 DMS Plant Feed

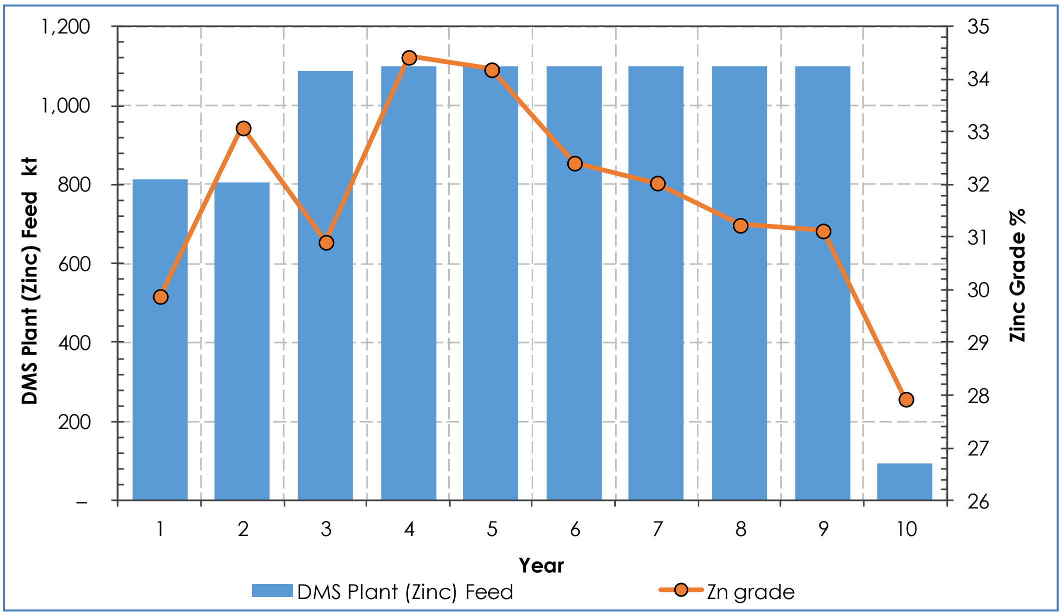
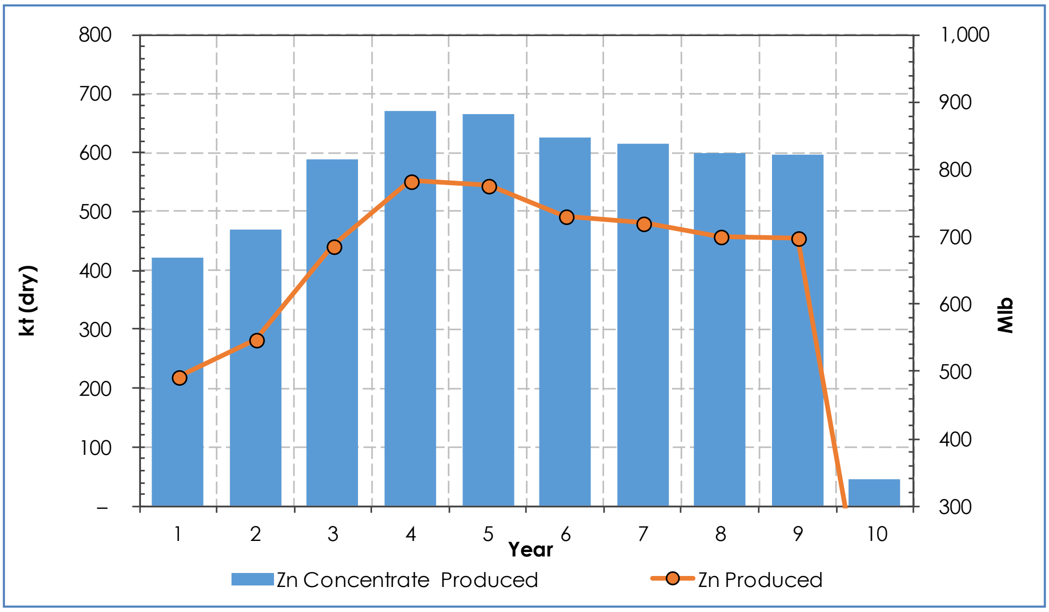


Figure 17.3 Zinc Concentrate and Metal Produced



The Kipushi mine plan includes copper plant feed of a total of 547 kt at 5.41% Cu that is planned to be treated under a tolling arrangement. Total copper concentrate produced is 106 kt (dry) at 25% Cu. The planned copper processing schedule is summarised in Table 17.2.

Table 17.2 Copper Processing Schedule

	Unit	Total	Schedule Year									
			1	2	3	4	5	6	7	8	9	10
Copper Feed Tonnes Processed												
Copper Plant Feed	**kt**	**547**	**17**	**166**	**171**	**50**	**36**	**22**	**14**	**72**	**–**	**–**
NSR DMS	$/t	64.37	51.85	68.64	34.55	60.44	37.29	104.86	102.43	125.11	–	–
NSR Cu	$/t	203.78	294.72	252.42	189.10	169.27	178.53	177.80	153.65	159.06	–	–
Zn grade	%	8.02	6.54	8.58	4.56	8.03	4.55	13.22	12.25	14.64	–	–
Cu grade	%	5.41	7.80	6.68	5.01	4.61	4.73	4.71	4.07	4.21	–	–
Copper Concentrate												
Cu Recovery	%	90.00	90.00	90.00	90.00	90.00	90.00	90.00	90.00	90.00	–	–
Concentrate Produced	kt (dry)	106	5	40	31	8	6	4	2	11	–	–
Cu grade	%	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	25.00	–	–
Cu Produced	kt	26.61	1.20	9.96	7.72	2.06	1.52	0.92	0.51	2.73	–	–

Planned Copper plant feed, copper concentrate and quantity of metal produced are shown in Figure 17.4 and Figure 17.5.

Figure 17.4 Copper Plant Feed

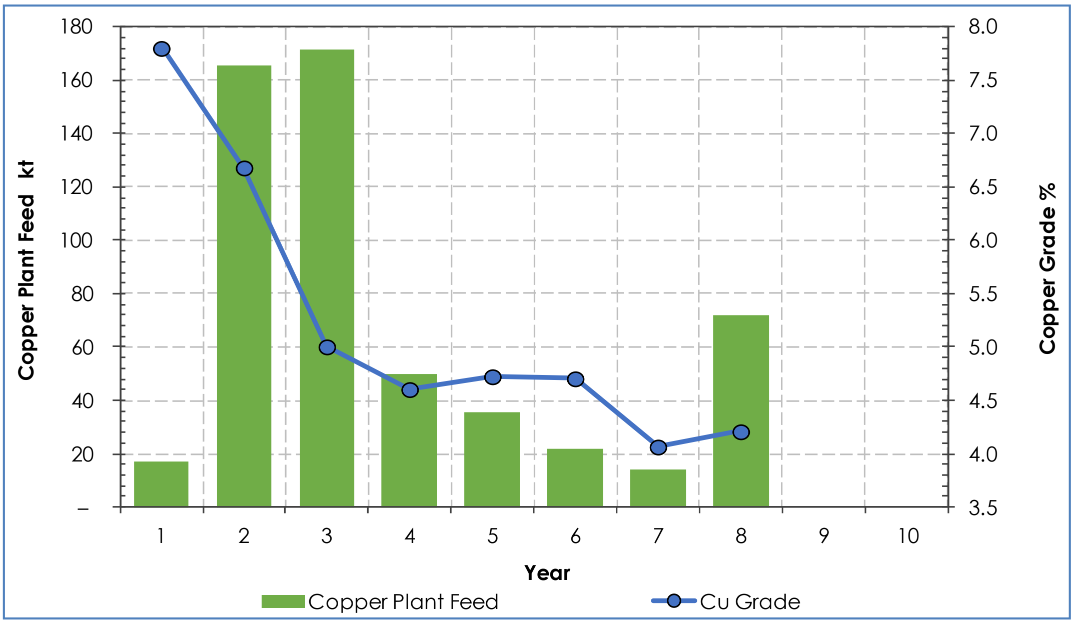
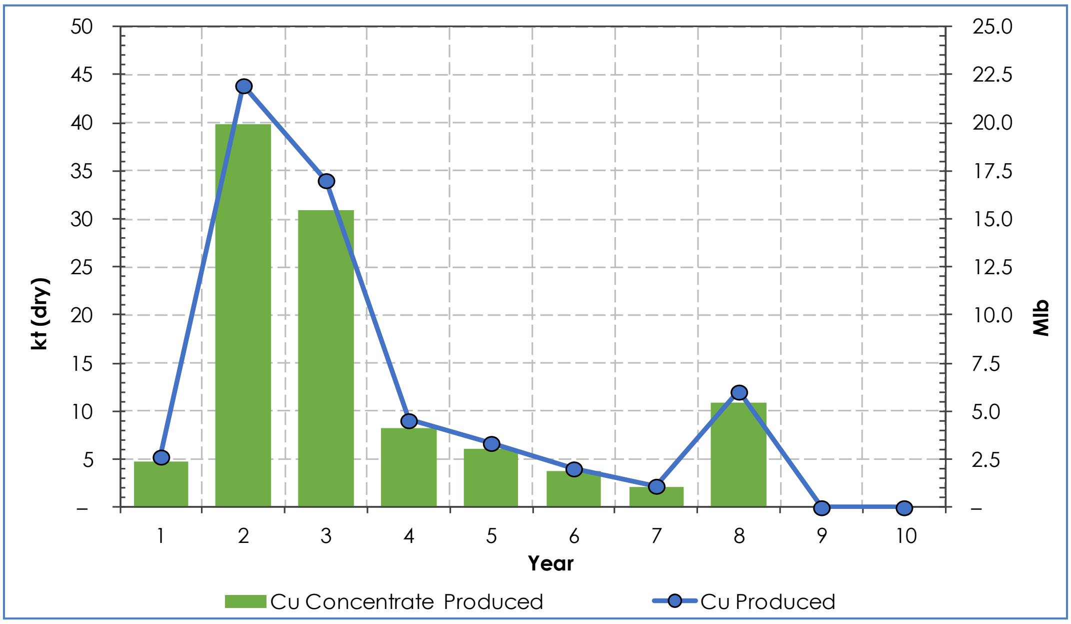


Figure 17.5 Copper Concentrate and Metal Produced



18 PROJECT INFRASTRUCTURE

The Kipushi Zn-Cu Project includes surface mining and processing infrastructure, concentrator, offices, workshops, and a connection to the national power grid. Electricity is supplied by the DRC state power company, Société Nationale d'Electricité (SNEL), using two transmission lines from Lubumbashi. Pylons are in place for a third line. The surface infrastructure is owned by Gécamines, KICO has entered into an agreement to use the surface rights on the Kipushi Zn-Cu Project to the extent required for its operations.

18.1 Site Infrastructure

It has been assumed that the existing buildings and facilities at the site will be used with new facilities constructed for the DMS plant. The overall proposed site layout is shown in Figure 18.1.

A dry DMS tails stockpile area and fine Spiral tails settling dams are planned to be constructed to dewater the fine Spiral tails prior to it being reclaimed and combined with the coarse DMS tailings and used as underground backfill.

Waste rock from underground, hoisted via Shaft 5, is planned to be transferred and dumped into the historical open pit.

The services planned to be provided to (and from) the DMS plant include water supply (process, potable, raw, gland service and fire where required), air services (compressed and instrumentation air), power supply (permanent and construction power), overland conveying, the tailings disposal pipeline to the tailings deposition point. In addition, allowances have been made for plant vehicles and general spares.

The process plant site is readily accessible by a well-maintained tarred road and is within close proximity of the national power grid.

The town of Kipushi is located approximately 30 km from the Lubumbashi International Airport and flights are scheduled to and from South Africa several times a week.

18.1.1 Electrical

Electricity is supplied by the DRC state power company, Société Nationale d'Electricité (SNEL) with an existing purchase agreement for the supply of 34 MWhr/year. The existing surface electrical yard located at the Kipushi mine is operating and is expected to only require some refurbishment and/or maintenance and the construction of associated distribution substations in the relevant processing plant areas. A secondary substation and the 850 mRL underground substation is expected to require upgrading and installation of a new substation close to mining area, possibly at 1,220 mRL, to support mining.

18.1.2 Water

An abundant supply of process water from the underground dewatering operations is expected to provide adequate water for processing and mining operations.

Figure 18.1 Overall Proposed Site Layout

Figure by Ivanhoe, 2016.

18.2 Concentrate Transport and Logistics

18.2.1 Rail Transport

Rail systems in the DRC are owned and operated by La Société Nationale des Chemins de Fer du Congo (SNCC). This includes the Kipushi Station and connecting rail line from Kipushi to Manama and through to the Zambian border at Ndola. Grindrod Limited is a leading and experienced freight services, shipping and financial services operator in Southern Africa, listed on the JSE Securities Exchange. Grindrod, working with Ivanhoe, has established a preliminary rail solution to the export of bulk zinc concentrate from the Kipushi mine. Ivanhoe is working with Grindrod to advance discussions with SNCC regarding the concession from Kipushi to Manama. This rail solution forms the basis of the concentrate transport for this Kipushi 2016 PEA.

18.2.2 Kipushi Station Rail Infrastructure

The existing Kipushi Station is expected to require significant refurbishment, with the addition of sufficient rail capacity to allow two full trains and the ability for locomotives to transfer from the incoming train to the outgoing train. Approval from SNCC to operate the SNCC infrastructure, including the Kipushi Station, as a private use facility will be required. It is anticipated that this could be granted by the required time to meet the expected Project schedule. It is assumed that the rail contractor will gain this and any other approvals necessary to allow the rail solution to come to fruition. The railway infrastructure in the Kipushi Station yard should consist of three lines with a combined length of 3,600 m. The anticipated dedicated zinc concentrate train length of 76 wagons plus locomotives is 1,100 m.

The assumption is that the rail contractor is to allow the incoming train arriving at the Kipushi Station on one dedicated siding and the locomotive to transfer from this incoming train to the outgoing train on a connected siding and leave for the trip to the export port. The incoming train wagons are then expected to be unloaded of containers to the bonded container yard, and loaded concentrate containers loaded onto the train wagons ready for the next trip to the export port.

18.2.3 Rail Kipushi to Durban

The proposed export route for zinc concentrate is to utilise containerised rail from Kipushi to the export port of Durban, South Africa. This route is planned to utilise the SNCC network from Kipushi to Ndola, then connecting to the North-South Rail Corridor from Ndola to Durban. The North–South Rail Corridor is described in Figure 18.2. The SNCC rail network in the Haut-Katanga Province is shown in Figure 18.3.

The Kipushi to Manama branch line, is assumed to require significant refurbishment of the whole 30 km. From Kipushi to Manama to Sakania trains would be run by the rail contractor under an Access Agreement with SNCC. From Sakania to Durban trains run under the joint operating centre (JOC), a partnership between Transnet, Swaziland Railway Authority and Mozambique’s port and rail regulators. Transnet can haul to Richards Bay or the Durban Container Terminal.

The distances of the rails lines between Kipushi and Durban are shown in Table 18.1. The total distance is 3,181 km with an expected transit time of 12 days.

Figure 18.2 DRC to South Africa North–South Rail Corridor

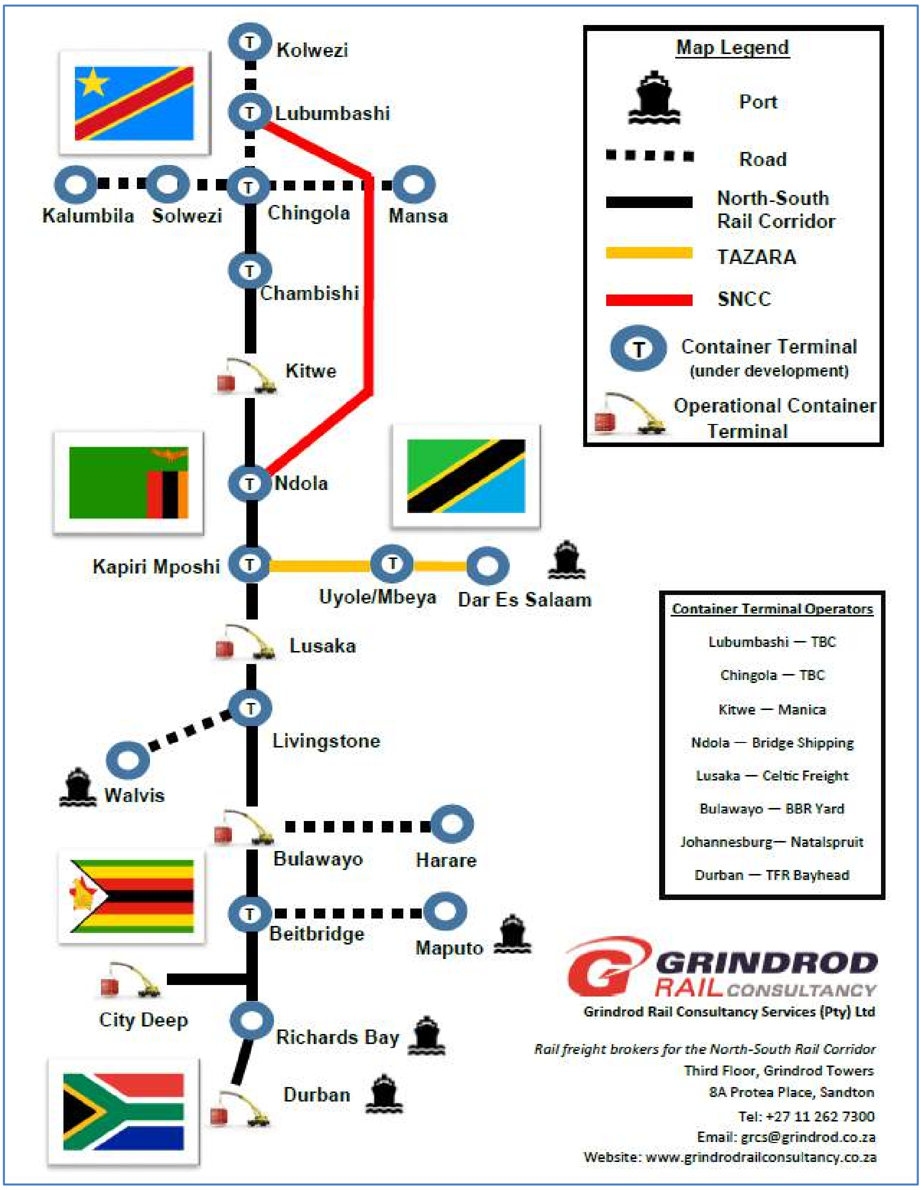
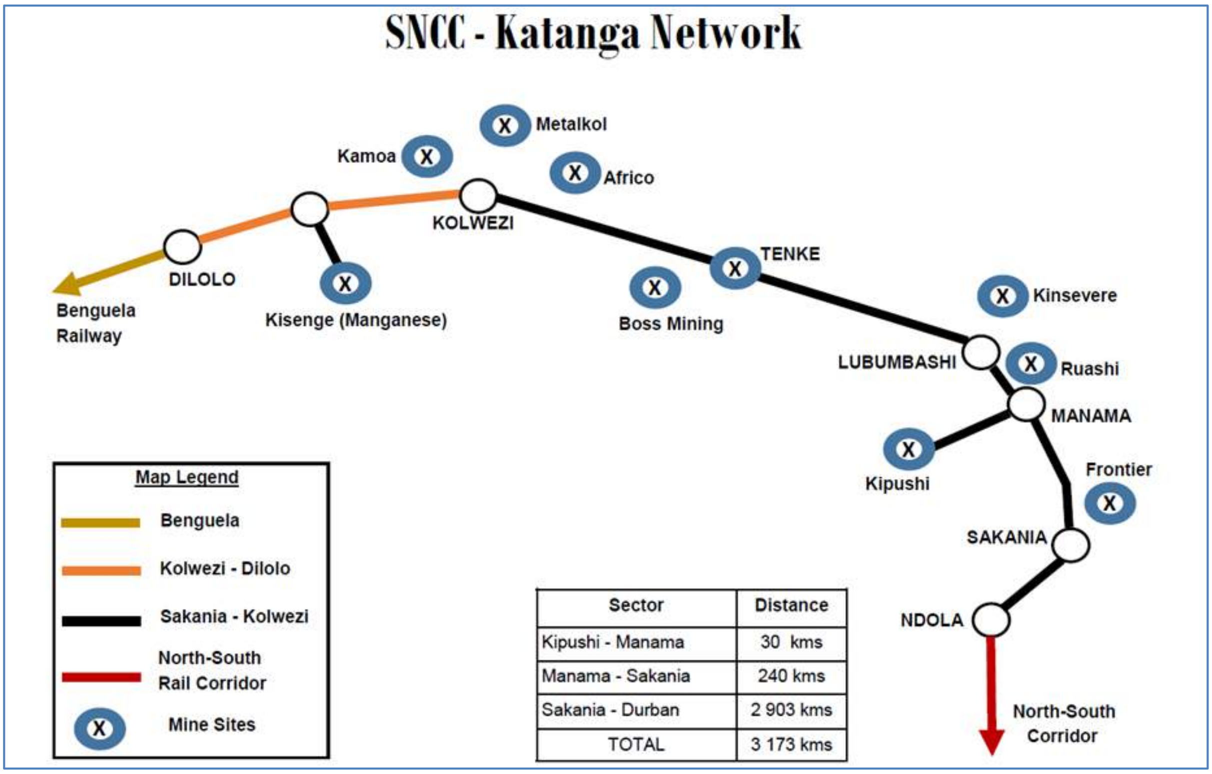


Figure by Grindrod, 2016.

Figure 18.3 SNCC Katanga Rail Network



Sector	Distance
Kipushi - Manama	30 kms
Manama - Sakania	240 kms
Sakania - Durban	2 903 kms
TOTAL	3 173 kms

Figure by Grindrod, 2016.

Table 18.1 Rail Line Distances between Kipushi and Durban

Sector		Distance (km)
From	**To**	
Kipushi	Manama	30
Manama	Sakania	240
Sakania	Victoria Falls	794
Victoria Falls	Beitbridge	815
Beitbridge	Durban	1,302
Total		**3,181**

The North–South Rail Corridor from Sakania to Durban via Zimbabwe has a capacity of 5 Mtpa (equivalent to approximately four trains per day in each direction). Whilst the railway infrastructure is old and has lacked significant investment for over 50 years, average speeds of around 30 kmph are still achieved on a daily basis. Unlike trucks, trains continue to travel at night and have limited delays at international borders. This makes the rail solution very competitive with road transport.

The Southern African Development Community (SADC) is developing a regional rail master plan that focuses on the development of multiple rail options for SADC countries. The SADC rail network is shown in Figure 18.4.

The NEPAD Business Foundation (NBF) is a non-profit organisation whose mandate is to support the accelerated development of infrastructure on the African continent through regional integration, increasing the efficiency of the rail, the growth of trade and sustainable development particularly infrastructure in the SADC region. NBF completed a Condition Assessment Report, in February 2015, on the North–South Rail Corridor which detailed the condition of the current rail infrastructure from Lubumbashi (DRC), through Ndola (Zambia), Zimbabwe, to Richards Bay or Durban (South Africa).

The major piece of infrastructure on the North–South Rail Corridor is the Victoria Falls Bridge at Livingstone. This bridge, which may need substantial repairs at some time in the future, remains a risk to this transport route to the sea. In the event of a possible closure of the Victoria Falls Bridge the proposed rail freight from Kipushi could be diverted along the Tazara Railway to Dar es Salaam see Figure 18.4. An alternative is the route westwards via Kolwezi to the Angolan port of Lobito. Completion of this route between Dilolo and Kolwezi is as yet incomplete but the Lobito to the Luau-Dilolo is new and in excellent condition with significant capacity.

Figure 18.4 SADC Rail Network

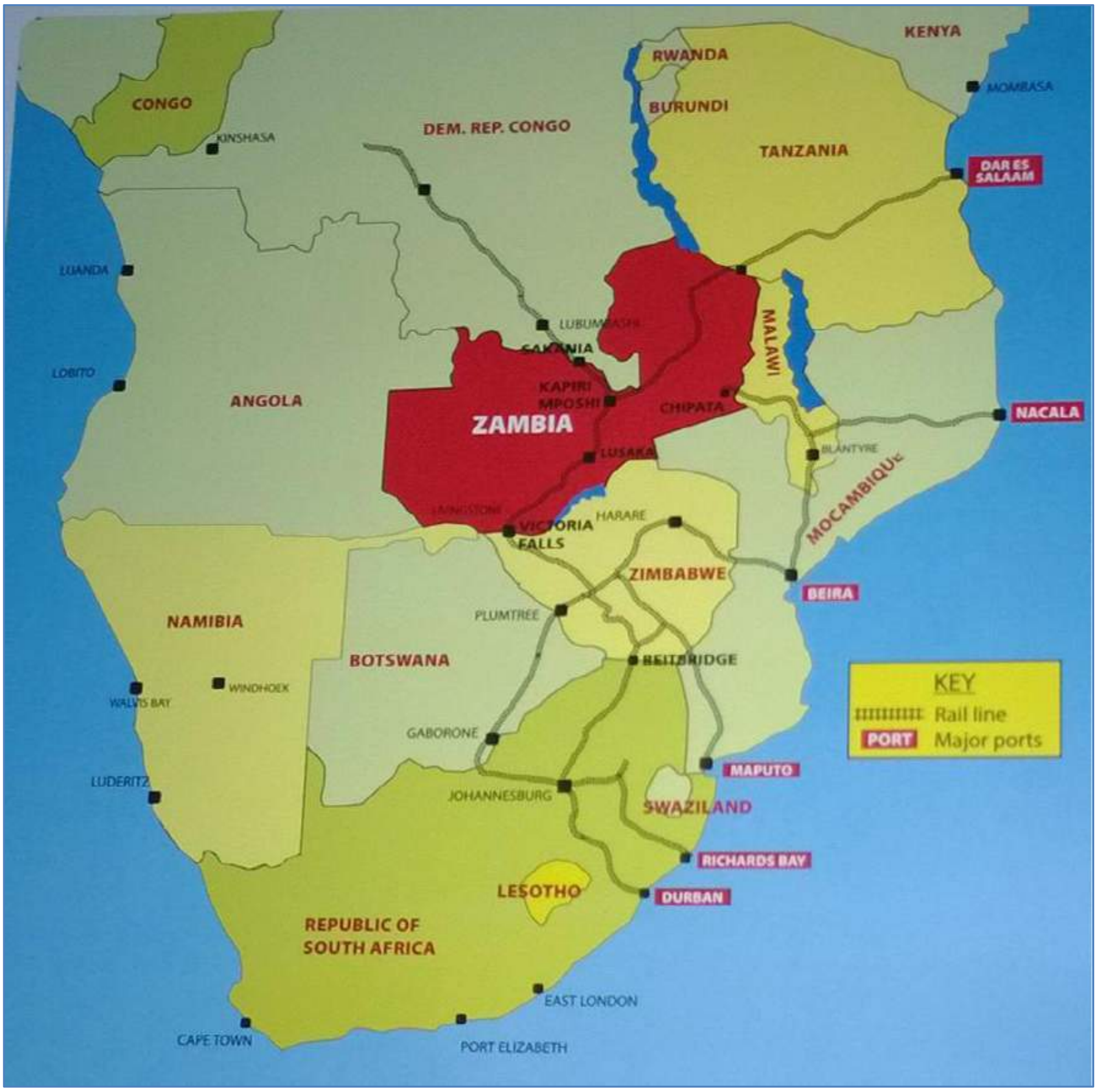


Figure by Grindrod, 2016.

18.3 Ocean Shipping / Freight

18.3.1 Port of Durban

The Port of Durban is Africa's largest and busiest port, well located to service markets in Europe, Asia, East Coast South America (ECSA), India, and even North America. Apart from ECSA and India, all of the identified likely destinations for the Kipushi zinc and copper concentrates would see average sail times in the 30–40 day range.

The Port of Durban is a deepwater facility with average draft at the berths in the +/-10.0 m range making it fully capable of handling Handysize vessels; those most commonly used in the transportation of base metal concentrates.

Several companies offer bulk storage and handling services for a variety of goods including mineral and metallurgical products.

18.3.2 Shipping Market Outlook

The expected ongoing growth in aggregate demand for seaborne bulk commodities transportation should support a relatively healthy environment in the medium-term although the ongoing construction of new shipbuilding capacity in China and other nations is expected to temper any supply gap in the coming years' thereby potentially limiting upside to rates.

During the period 2016–2020 the market is expected to be more stable than the last decade, with continued growth in the movement of bulk commodities to meet global demand but with a ship building industry more capable of absorbing the increased requirements.

As Africa's busiest port, the Durban-Europe and Durban-Asia trade route is well plied. Accordingly, for main European ports and for ports in Japan, Korea, and China, good competition for cargoes from ship owners can be expected.

The cost to deliver concentrates to the two Indian smelters is more expensive compared to other routes. The parcel service from Durban to India has a lack of combination cargoes for this area and limited outbound opportunities mean rates are based solely on inbound-India cargoes with no outbound. If combinations were possible then better rates may be attainable.

Although there is little evidence of a recovery in ocean freight rates, prevailing rates are not expected to be sustainable as returns on investment will simply not support the required additions to the fleet. Assuming that fleet modernisation can be managed in an orderly fashion to ensure that adequate capacity is available going forward, ocean freight rates are expected to gradually trend upward.

18.4 Additional Transportation Issues

18.4.1 Container Transport

Based on bagged concentrate at mine being loaded into export containers handling losses are expected to be zero. Based on the movement of concentrate from the mine to the receiving smelter the handling points would be as set out in Table 18.2.

Table 18.2 Concentrate Handling Points

1	Bagging shed to container yard. Individual bags weighed and identified.
2	Loading concentrate bags into export containers. container number, bag weights and ID recorded.
3	Export containers onto rail and transported to Port of Durban
4	Export containers stored at port until loaded on ship. Container number recorded.
5	Export containers loaded on ship and shipped to smelter port
6	Export containers unloaded from ship and transported to smelter for unloading

18.4.2 Insurance

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, or NIV).

18.4.3 Representation

Inspection / representative 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 loadport to monitor terminal activities.

18.4.4 Shipping Conclusions

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 NSRs between the alternative destinations. The freight component will be critical to ensure maximisation of NSRs.

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.

19 MARKET STUDIES AND CONTRACTS

19.1 Zinc Market Overview

For several years the zinc market has faced the prospect of significant impending mine closures with limited apparent replacement capacity. The deficit shocks expected to be created by these closures has been slow to emerge due to a combination of:

- Slower metal demand growth associated with a weaker global economy,
- Higher than expected mine output from other sources, and
- The quasi-regular appearance on the exchanges of large quantities of unreported stocks.

The global demand for refined zinc (Table 19.1) has grown by close to 2.5 Mt over the past decade. As with most other metals, China has become the biggest factor in the market, accounting for roughly half of global consumption in 2015, up from less than a third a decade ago. Future zinc demand is expected to remain steady, constrained only by the potential shortage of mine supply towards the end of the decade. 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 (kt Zinc)

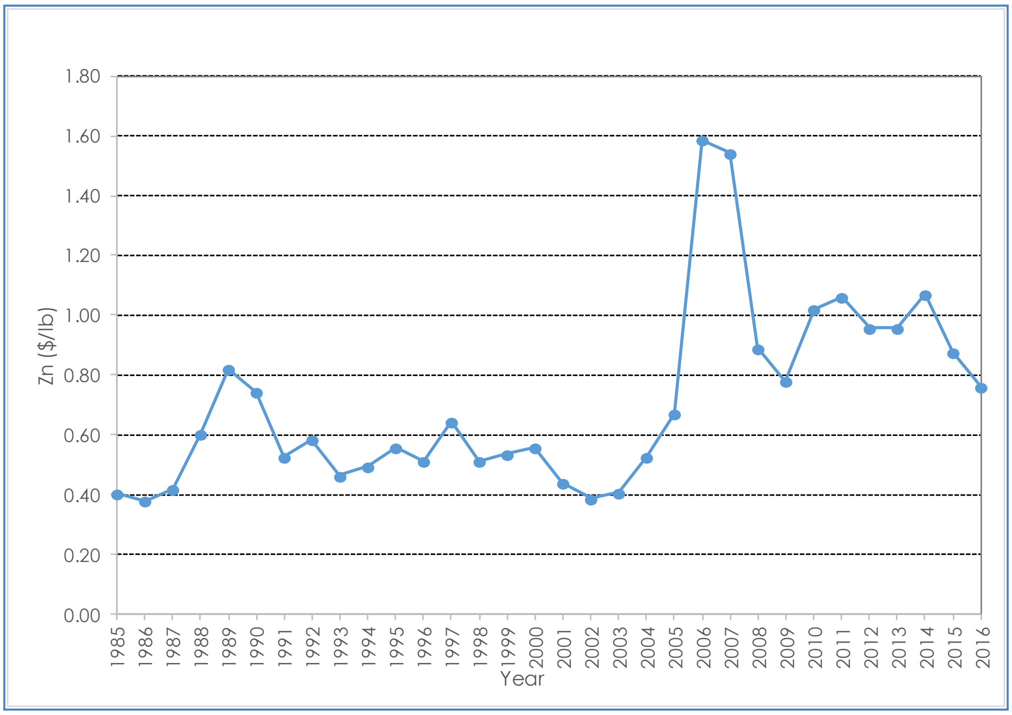
	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019
Zn Supply	11,280	12,896	13,066	12,630	12,873	13,304	14,102	14,734	14,935	15,421	15,452
Global Demand	10,757	12,702	12,696	12,285	12,933	13,536	14,124	14,786	15,204	15,559	15,730
Of which: China	*4,500*	*5,453*	*5,458*	*5,243*	*5,703*	*6,204*	*6,662*	*7,167*	*7,482*	*7,763*	*7,899*
Surplus (Deficit) before Glencore Announcement	523	194	370	345	(60)	(232)	(22)	(52)	(269)	(138)	(278)
Glencore Cutbacks (adj)	–	–	–	–	–	–	(75)	(400)	(25)	–	–
Surplus (Deficit) after Glencore Announcement	523	194	370	345	(60)	(232)	(97)	(452)	(294)	(138)	(278)

(Source: Wood Mackenzie, RBCCM, ILZSG. Glencore)

Mine closures at MMG Ltd's Century mine in Australia (466 ktpa), and Vedanta Resources Plc's Lisheen mine in Ireland (133 ktpa) combined with the previous closure in 2013 of Glencore / Xstrata's Perseverance and Brunswick mines (total production over 300 ktpa) has permanently removed in excess of 900 ktpa mined Zinc production from the market, equivalent to approximately 4.5% 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 2006/2007, the zinc price traded within a wide range of around $600/t ($0.27/lb) to $2,000/t ($0.97/lb) but averaged less than $1,100/t ($0.50/lb) (Figure 19.1).

Figure 19.1 Zinc Price (1985–2016) ($/lb)



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).

In April 2016, zinc inventories dropped to 413,250 metric tonnes, the lowest levels since August 2009, as supply cuts and the closing of the Century and Lisheen mines in 2015 has resulted in the continued drawdown of exchange inventories.

19.1.1 Market Factors

Two major factors could have a bearing on the zinc concentrate market:

- Market Influence of China
- 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.

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, and
- Increasing environmental and social challenges.

Global refined production however 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 the west for the balance of this decade; any new western capacity is expected to be limited to brownfield expansions and debottlenecking.

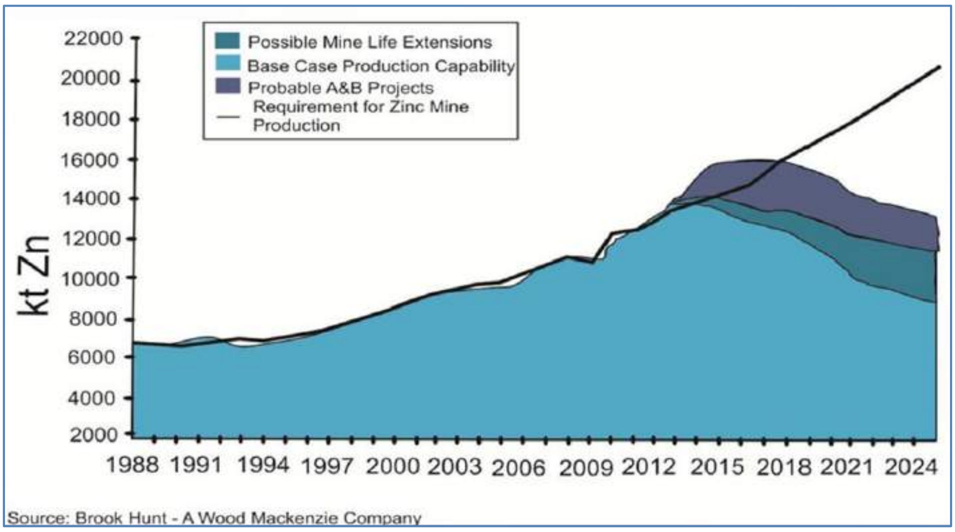
Over the past decade, in an effort to satisfy growing domestic zinc metal demand, Chinese smelting capacity has increased four-fold since 2000, rising from an annual refining capacity of approximately 1.9 Mt of zinc to over 8 Mt in 2015.

Chinese, and to a lesser extent Indian, smelting companies may continue to expand capacity in an attempt to match growing domestic metal demand. Although it is expected that sufficient zinc refining capacity will be available to meet demand for metal, the more relevant issue remains whether mine supply can in fact rise to meet this demand.

19.1.3 Projected Zinc Concentrate Supply/Demand Balance

A deficit in zinc concentrate supply continues to be projected and it is not expected that enough new production will be on line before the end of the decade to compensate for the large scale attrition. Concentrate shortfalls are expected to be severe, which should translate into significantly reduced metal supply. While improving market fundamentals will support new mine developments it is not expected that sufficient production can be brought on stream much before the end of the decade to significantly reverse this projected trend. Accordingly, a serious long-term supply gap is expected to emerge which can only be reversed if prices rise to incentivise development of these currently uncommitted projects (Figure 19.2).

Figure 19.2 Impending Zinc Supply Gap



Source: Brook Hunt - A Wood Mackenzie Company

19.1.4 Treatment Charge Outlook

The Kipushi 2016 PEA assumes that zinc concentrate will be sold at industry standard terms. A long term concentrate treatment charge of $200/dmt concentrate, based on $2,200/dmt zinc price, has been assumed.

19.2 Kipushi Zinc Concentrate

19.2.1 Concentrate Quality Considerations

For smelters / 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. A comparison of an ideal zinc concentrate and the projected Kipushi Zinc concentrate characteristics is detailed in Table 19.2.

Table 19.2 Ideal Zinc Concentrate Specifications

Element	Ideal Content Range	Kipushi Zinc Concentrate
Cadmium	<0.3%	0.22%
Chlorine	<500 ppm	260.3 ppm
Copper	<0.5%	0.3%
Fluorine	<200 ppm	861.3 ppm
Iron	<6%	7.5%
Lead	<1%	2.1%
Mercury	<20 ppm	37.9 ppm
Moisture	7%–9%	8%
Silica	<2%	1.2%
Sulphur	28%–33%	29.3%
Zinc	52%–58%	55.4%

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.

Penalties 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 / 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.

The life-of-mine average annual production for the Kipushi Zn-Cu Project is projected to be approximately 530,000 dmt zinc concentrates with grades as set out in Table 19.3.

Table 19.3 Projected Kipushi Zinc Concentrate Grade

	Zn (%)	Pb (%)	Fe (%)	Ca (%)	Si (%)	Cu (%)	Mg (%)	S (%)
Final Zn Concentrate	55.4	2.1	7.5	0.7	1.2	0.3	0.3	29.3

	Au (ppm)	Ag (ppm)	Ge (ppm)	Cd (ppm)	Sb (ppm)	Hg (ppm)	As (ppm)	Cl (ppm)	F (ppm)
Final Zn Concentrate	0.3	33.4	82.7	2,159	9.8	37.9	874.2	260.3	861.3

Based on the expected analysis there are no material quality issues foreseen with these 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 a 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.
- 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 annual production average of 530 kt concentrate, the Kipushi Zn-Cu 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 direct 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 is 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 Zn-Cu Project while taking into consideration prevailing market conditions at the time contract discussions are entered into.

19.3 Copper Concentrate

Copper production is assumed to be toll treated. There is potential to sell copper concentrate to smelters in Zambia and or merchants where more favourable terms may be possible.

The Kipushi 2016 PEA assumes that copper concentrate will be sold at industry standard terms. The current market outlook is for a long term concentrate treatment charge of $90/dmt concentrate and refining charge of $0.09/lb of copper. This has been used in the economic analysis for the Mineral Reserve. The base case analysis for the Kipushi 2016 PEA assumes a copper price of $3.00/lb, this is consistent with long term estimates and pricing used in other published studies.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

The climate in the Kipushi area is humid subtropical hot summer climate that is mild with 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 no available fertile agricultural land available for new allocation.

Although there is a significant environmental legacy from previous operation of the mine, Gécamines have been exonerated by the DPEM, and there is no legal obligation for KICO to undertake rehabilitation.

Sustainability for the Kipushi Zn-Cu 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, community health initiatives including FIONET, and support to local suppliers to the mine based in Kipushi. Although there is considerable small scale agriculture in the impact area, due to the natural infertility of the soil verse the cost of tillage and fertilisers, this is seen as not self-sustainable.

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.

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.

On 24 January 2016, the licenses for PER12234, PER12249, and PER12250 were allowed to lapse at the Cadastre Miniere (CAMI) as they are not necessary for the reject from the planned zinc processing plant. Any tailings or waste rock produced from the zinc beneficiation will be used as mine backfill or stored in the former open pit excavation.

20.2 Force Majeure Condition

The legal condition of force majeure on PE12434 was applied mid-2011 as a result of the mine flooding, following the failure of the main underground pumping station at approximately 1,200 mRL in Shaft 5.

The condition of force majeure suspends some of the regulatory requirements of environmental reporting and discounts on some regulatory services, including SNEL invoicing for electricity supply, and BECT inspections of conveyances.

Force majeure is lifted on notification to the Mines Ministry that the conditions which caused the implementation of force majeure are corrected.

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 Direction de Protection de L’Environnment Miniere (DPEM) in August 2011 with the conclusion:

> “…Given that Gécamines has run its exploitation activities while considering the reduction and the rehabilitation on the perimeters of the PER n°12234 12349 12350, and the PE12434 on assignment to KICO Sprl, Gécamines should be freed from the environmental obligations on these perimeters except the part used for treatment by the CMSK and the retention basin it uses.
>
> So, the Kipushi Corporation Company will be responsible of damages it causes on the environment once it will be installed in the perimeter and must take already necessary measures to prepare an environmental plan relative to its activities and allowing him to encounter negative impacts of its exploitation.” (Translation from the original French version).

Therefore, KICO is only responsible for the environmental impacts going forward, although there may be a social obligation to mitigate some of the historical impacts, including fugitive dust and particularly on closure of the new operations at life-of-mine.

20.4 Golder Associates Africa ESHIA Baseline Study

Golder Associates Africa has completed several reports on the Kipushi Zn-Cu Project, including:

- TSF Trade-off Study for the future mine tailings disposal facilities, November 2013
- 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 TSF3.

- 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.
- 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.
- Survey of health care facilities.
- Survey of Employee’s residence locations and proximity to medical service providers.

20.6 KICO Community and Social Activities

KICO has 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, small scale agriculture test plots.

- 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.

20.7 Tailing Management and Disposal

There is expected to be two tailings streams in the proposed process plant; coarse DMS tailings and fine Spiral tailings.

The coarse DMS tailings, nominally –20+1 mm, material is expected to be picked up by FEL and placed into trucks. The trucks will transport the material to either; Shaft 3 or the coarse tailings stockpile area if there is insufficient capacity at Shaft 3.

The fine Spiral tailings material, nominally –1 mm, after thickening are planned to be pumped to one of two settling dams. The settling dams, one in use and one being reclaimed, are a temporary storage and dewatering location for the fine tailings.

The settled fine tailings are expected to be reclaimed and combined with the coarse DMS tailings at Shaft 3 where all the tailings are planned to be sent underground and used as backfill.

20.8 Water Management

20.8.1 Surface Water

The catchments in the vicinity of the Kipushi Mine are shown in Figure 20.1. The mine is located in the upper reaches of the Kipushi Catchment with the mine tailing storage facilities (TSFs) located in the middle reaches of the Kipushi River. The Kanie Mesha River joins the Kipushi River from the north about 3 km downstream of the TSFs. The Kipushi River flows east for another 1 km before it joins the Kafubu River. The Kamalenge River flows in 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. 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.

The Kafubu River flows east towards Lubumbashi and water is extracted from the river to supply Lubumbashi.

Figure 20.1 Location and Extent of the Surface Water Catchments in the Vicinity of Kipushi Mine

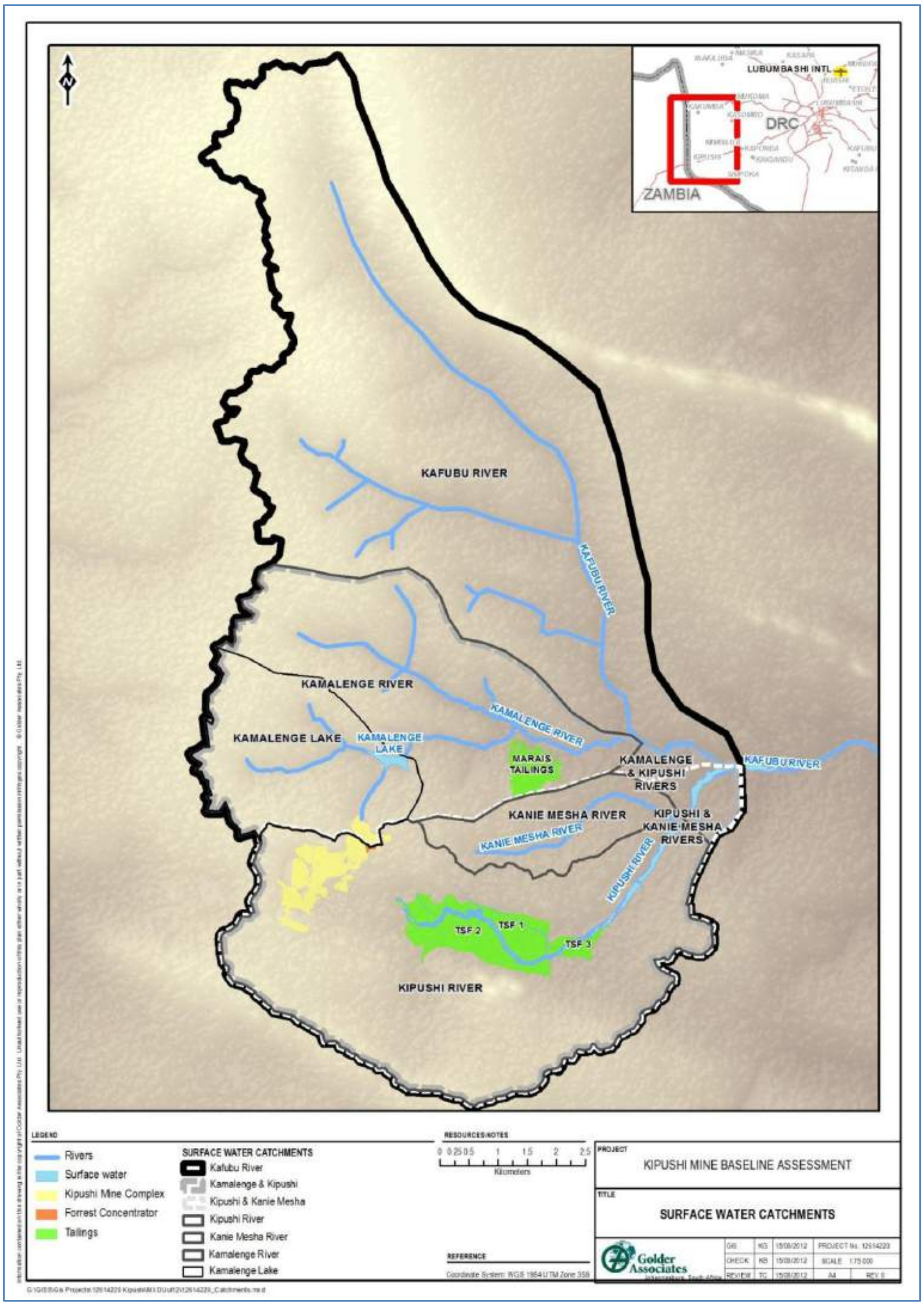


Figure by Golders, 2012.

20.8.2 Current Stormwater Management System

The stormwater management system around Kipushi Mine has been upgraded by KICO. The major components of the current stormwater system are shown in Figure 20.2.

There is stormwater diversion channel which flows to the south of TSF1 and TSF2. The drain starts near the Water Storage Area and discharges onto TSF3 which flows into the Kipushi River. This drain prevents run-off from the catchments to the south of the mine from entering the mine area.

Water from underground is pumped out and then discharged into the main stormwater drain which discharges into the channel draining through the town. The stormwater run-off from the mine complex and the villages flows onto TSF1. The Kipushi River has re-established itself to flow over the TSFs to discharge to the Kipushi River at the outlet works at the TSF3 dam wall. The TSF3 dam wall has a spillway to take typical flows draining across the TSF complex however it may not be suited to extreme flood events.

Figure 20.2 Current Stormwater System at Kipushi Mine

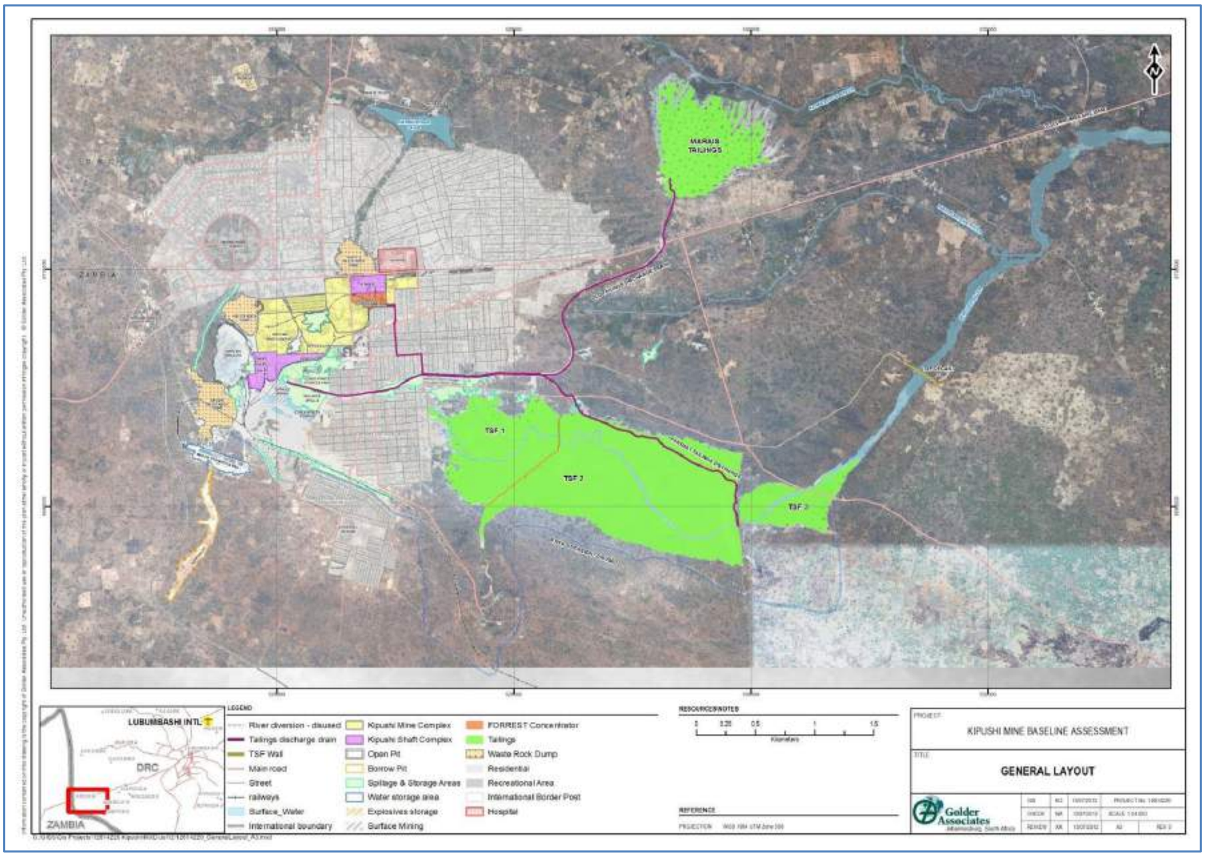


Figure by Golders, 2012.

20.9 KICO Plans Going Forward

The next steps in the environmental management of the project include:

- Ongoing monitoring of surface and groundwater, air quality and climate to meet regulatory reporting requirements.
- Completion of a regulatory Environmental Impact Statement and Environmental Management Plan.
- Longer term livelihood change for artisanal miners, focused on SMEs to provide services to the Kipushi mine development.
- Identify other permitting requirements.
- Prepare detailed closure plan.

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 include a contingency of 30%. Additional detail work is required to define the costs. All monetary figures expressed in this report are in US dollars (US$) unless otherwise stated.

21.1 Capital Cost

The total Project direct capital cost estimates are shown in Table 21.1.

Table 21.1 Estimated Total Project Capital Cost

Description	Pre-Production ($M)	Sustaining ($M)	Total ($M)
Mining:			
Rehabilitation	111	–	111
Underground	52	84	136
Capitalised Mining Operating Costs	37	–	37
Subtotal	**200**	**84**	**284**
Process and Infrastructure:			
Process and Infrastructure	32	6	38
Subtotal	**32**	**6**	**38**
Closure:			
Closure	–	20	20
Subtotal	**–**	**20**	**20**
Indirects:			
EPCM	29	2	32
Capitalised G&A and Other Costs	60	–	60
Subtotal	**89**	**2**	**92**
Other:			
Owners Cost	29	2	32
Capital Cost Before Contingency	**350**	**115**	**465**
Contingency	58	5	63
Capital Cost After Contingency	**409**	**119**	**528**

Notes:

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.

Mining capital cost estimates are shown in three parts: Rehabilitation, Underground and Capitalised Operating Costs. Rehabilitation costs include the current KICO budget for the mine to re-establish Shaft 5 and complete the mine pumping system as well as other works. The Underground capital cost estimates include the mining mobile equipment, fixed equipment (crusher, electrical, rail, construction of chutes and passes), waste and access development allocation. Mining operating cost estimates incurred prior to plant start-up have been shown as Capitalised Mining Operating Costs.

The estimated capital cost for the process plant accounts for all items with the following battery limits.

- Crushed material onto new conveyor connecting Shaft 5 to the process plant ROM.
- Discharge of tailings slimes into a slimes tailings facility.
- Pipeline for copper tailings from copper concentrator to slimes tailings facility.
- Trucks loaded with coarse tailings at the process plant, but including installation of roadworks to the intermediate stockpile near Shaft 3.
- Trucks loaded with bagged concentrate at the bagging plant, but includes installation of roadworks to the container loading / storage yard adjacent to Kipushi Station.

The estimated capital cost is made up of preliminary proposals, DMS Process Plant, and in-house estimates based on understanding of project requirements and site visits as at September 2015.

The process and infrastructure capital cost estimate is derived from a proposal from Bond Equipment for the DMS process plant with a design limit of 150 tph for the DMS circuit and 20 tph for the spiral circuit at 85% availability or 1.1 Mtpa. All other estimates are derived for estimates of distance between key locations and in-house or industry rates for a project located in DRC.

Unit rates for civils, earthworks and specified equipment are derived from recent projects in Africa. Factors for electrical, installation and steelwork are industry factors based on mechanical equipment for a plant of this degree of complexity. The concentrate bagging facility estimate is based on a semi-automated concentrate bagging plant to handle the tonnages estimated. The bagging facility is housed within a suitable building with conveyor and truck access.

EPCM and Owners costs have been estimated using factors of 15% on total capital for each. The capital cost estimates include a contingency of 30%.

Repair and refurbishments of the 34 km of track between Kipushi and Manama is anticipated to be undertaken by the rail contractor.

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 ($M)	5-Year Average	LOM Average
		($/t Milled)	
Site Operating Costs:			
Mining	536	58	54
Processing Zn and Cu (tolling)	87	10	9
General & Administration	120	11	12
Total	**715**	**79**	**75**

21.3 Concentrate Handling Cost

It has been assumed that a contractor will move the concentrate from the bagging facility to the container yard at Kipushi Station. An allowance of $0.20/t km for a total of $0.40/t concentrate for road haulage, including road maintenance has been assumed. The process operating cost includes the cost of loading the concentrate bags on trucks at the bagging plant.

A cost of $20/t concentrate has been allowed for bringing containers to Kipushi. An opportunity exists to backload imported items such as fuel and consumables at minimal freight cost, as empty containers will be returning to site daily. In practice it will be necessary to have an arrangement with the shipping lines to bring containers from the port to Kipushi. It is expected that there would be an agreement with several shipping lines and the trains would be scheduled so as to be despatched to each shipping line.

21.3.1 Concentrate Transport Costs

The operating costs for transport from Kipushi via Durban in South Africa to China (including all taxes) is estimated to total $249.61/t concentrate.

This estimate includes the following:

- Handling Mine Site to Kipushi Station
- Rail Transport DRC
- Rail Transport Zambia to South Africa
- Port Charges Durban
- Ocean Freight – Durban Port to Shanghai Containerised
- Logistics Agent Fees
- DRC Government Taxes, Levies, and Duties

22 ECONOMIC ANALYSIS

22.1 Production and Cost Summary

All monetary figures expressed in this report are in US dollars (US$) unless otherwise stated. The Kipushi Zn-Cu Project financial model is presented in 2016 constant US dollars, cash flows are assumed to occur evenly during each year and a mid-year discounting approach is taken. The key results of the Kipushi 2016 PEA are summarised in Table 22.1. The mining production forecasts are shown in Table 22.2 and forecast zinc and copper 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 22.1 Kipushi 2016 PEA Results Summary

Description	Unit	Total
Zinc Feed - Tonnes Processed		
Quantity Zinc Tonnes Treated	kt	9,394
Zinc Feed grade	%	32.15
Zinc Recovery	%	92.94
Zinc Concentrate Produced	kt (dry)	5,296
Zinc Concentrate grade	%	53.00
Copper Feed - Tonnes Processed		
Quantity Copper Tonnes Treated	kt	547
Copper Feed grade	%	5.41
Copper Recovery	%	90.00
Copper Concentrate Produced	kt (dry)	106
Copper Concentrate grade	%	25.00
Metal Produced		
Zinc	kt	2,807
Copper	kt	27
Key Cost Results		
Pre-Production Capital	$M	409
Mine Site Cash Cost	$/lb Zn	0.12
Realisation	$/lb Zn	0.44
Total Cash Costs After Credits	$/lb Zn	0.54
Site Operating Costs	$/t milled	74.77

Table 22.2 Mining Production Statistics

	Unit	Total LOM	5-Year Average	LOM Average
Zinc Feed - Tonnes Processed				
Quantity Zinc Tonnes Treated	kt	9,394	981	939
Zinc Feed grade	%	32.15	32.65	32.15
Zinc Recovery	%	92.94	93.14	92.94
Zinc Concentrate Produced	kt (dry)	5,296	562	530
Zinc Concentrate grade	%	53.00	53.00	53.00
Copper Feed - Tonnes Processed				
Quantity Copper Tonnes Treated	kt	547	88	55
Copper Feed grade	%	5.41	5.68	5.41
Copper Recovery	%	90.00	90.00	90.00
Copper Concentrate Produced	kt (dry)	106	18	11
Copper Concentrate grade	%	25.00	25.00	25.00
Metal Produced				
Zinc	kt	2,807	298	281
Copper	kt	27	4	3

Figure 22.1 Zinc and Copper Tonnes Mined

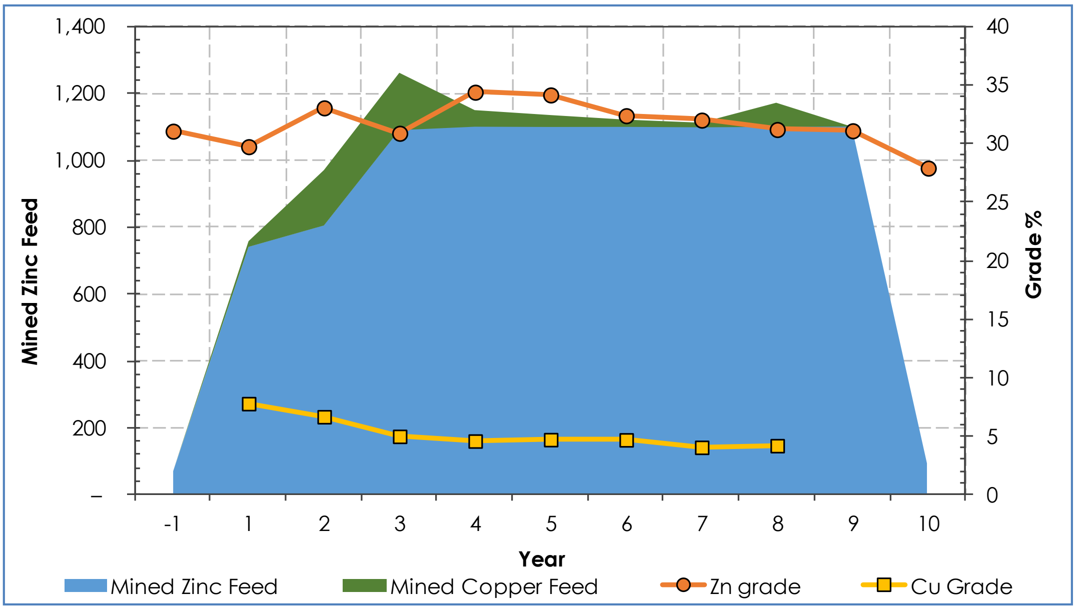


Figure 22.2 Zinc and Copper Tonnes Processed

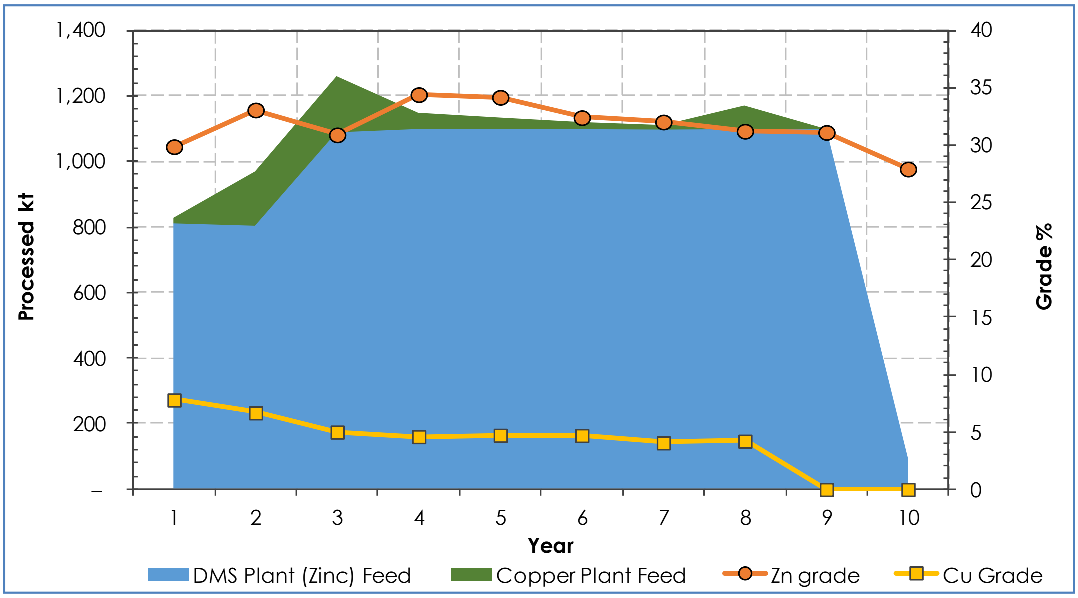
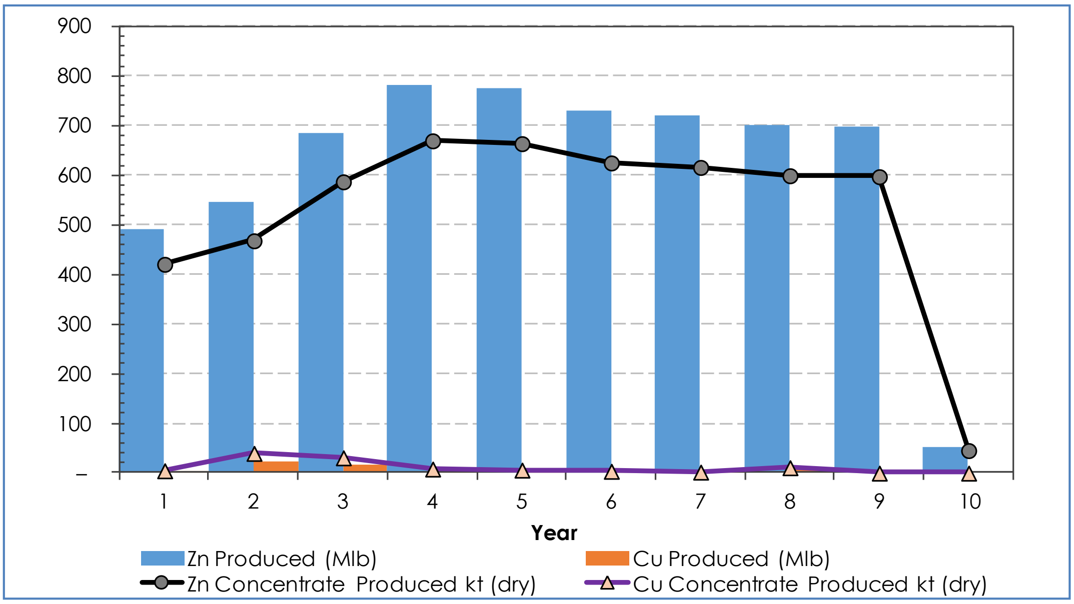


Figure 22.3 Concentrate and Metal Production



22.2 Project Financial Analysis

The estimated Mine site cash costs are shown in Table 22.3. Total estimated cash costs after credits for the first five years of production are $1,174/t zinc and the average for the life of the mine is $1,180/t zinc. 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 $533M
- After-tax internal rate of return (IRR) is 30.9%
- After-tax project payback period is 2.2 years

Table 22.3 Estimated Cash Costs

Description	**5-Year Average**	**LOM Average**
	($/lb Zn)	
Mine Site Cash Cost	0.13	0.12
Realisation	0.45	0.44
Total Cash Costs Before Credits	**0.58**	**0.56**
Copper Credits	(0.04)	(0.03)
Total Cash Costs After Credits	**0.53**	**0.54**

The estimated revenues and operating costs have been presented in Table 22.4, along with the estimated net sales revenue value attributable to each key period of operation. The analysis uses price assumptions of $2,227/t Zn and $6,614/t Cu. 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 22.5.

Table 22.4 Estimated Operating Costs and Revenues

Description	**Total ($M)**	**5-Year Average**	**LOM Average**
		($/t Milled)	
Revenue:			
Gross Sales Revenue	**5,481**	**555**	**551**
Less Realisation Costs:			
Transport Costs	1,466	147	147
Treatment & Refining Charges	1,074	108	108
Royalties	198	20	20
Total Realisation Costs	**2,737**	**275**	**275**
Net Sales Revenue	**2,744**	**279**	**276**
Less Site Operating Costs:			
Mining	536	58	54
Processing Zn and Cu (tolling)	87	10	9
General & Administration	120	11	12
Total	**743**	**79**	**75**
Operating Margin ($M)	**2,001**	**201**	**201**
Operating Margin (%)	**37**	**36**	**37**

Table 22.5 Estimated Total Project Capital Cost

Description	Pre-Production ($M)	Sustaining ($M)	Total ($M)
Mining:			
Rehabilitation	111	–	111
Underground	52	84	136
Capitalised Mining Operating Costs	37	–	37
Subtotal	**200**	**84**	**284**
Process and Infrastructure:			
Process and Infrastructure	32	6	38
Subtotal	**32**	**6**	**38**
Closure:			
Closure	–	20	20
Subtotal	**–**	**20**	**20**
Indirects:			
EPCM	29	2	32
Capitalised G&A and Other Costs	60	–	60
Subtotal	**89**	**2**	**92**
Other:			
Owners Cost	29	2	32
Capital Cost Before Contingency	**350**	**115**	**465**
Contingency	58	5	63
Capital Cost After Contingency	**409**	**119**	**528**

Notes:

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 22.6. The discounted cash flow analyses use price assumptions of $2,227/t Zn and $6,614/t Cu. The prices are based on a review of consensus price forecasts from a financial institutions and similar studies that have recently been published. The key economic assumptions for the analysis are shown in Table 22.7.

The results of NPV sensitivity analysis to a range of zinc prices and discount rates is shown in Table 22.8. The estimated Cumulative cash flow is depicted in Figure 22.4 and a complete cash flow is provided in Table 22.9.

Table 22.6 Financial Results

	Discount Rate	Before Taxation	After Taxation
Net Present Value ($M)	Undiscounted	1,473	1,076
	5.0%	973	696
	8.0%	759	533
	10.0%	642	444
	12.0%	542	368
	15.0%	418	273
	18.0%	318	197
	20.0%	262	154
Internal Rate of Return	–	36.4%	30.9%
Project Payback Period (Years)	–	2.1	2.2

Table 22.7 Economic Assumptions

Parameter	Unit	Financial Analysis Assumption
Zinc Price	$/t	2,227
Copper Price	$/t	6,614
Zinc Treatment Charge	$/t concentrate	200.00
Copper Treatment Charge	$/t concentrate	90.00
Copper Refining Charge	$/t Cu	198.42

Table 22.8 After Tax Zinc Price Sensitivity – Discount Rates

Discount Rate	**Zinc ($/t)**						
	1,500	**1,750**	**2,000**	**2,227**	**2,500**	**2,750**	**3,000**
Undiscounted	-157	325	719	1,076	1,507	1,901	2,295
5%	-210	146	436	696	1,008	1,293	1,577
8%	-230	69	315	533	794	1,032	1,269
10%	-240	28	249	444	677	889	1,101
12%	-248	-7	193	368	577	767	957
15%	-258	-50	123	273	452	614	777
18%	-264	-84	66	197	351	491	631
20%	-266	-102	35	154	295	422	549

Figure 22.4 Cumulative Cash Flow

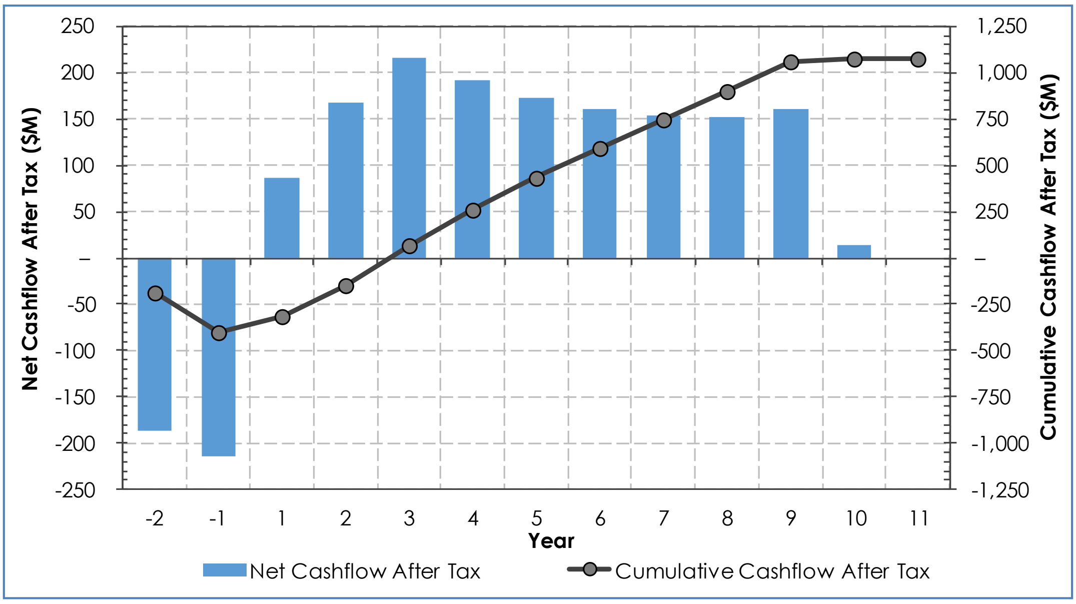


Table 22.9 Estimated Cash Flow

Description	Unit	Total	Year											
			-2	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**
Total Gross Revenue	$M	5,481	–	–	430	533	638	685	676	633	622	619	600	46
Total Realisation Costs	$M	2,737	–	–	216	259	314	343	339	318	313	309	302	23
Net Revenue	**$M**	**2,744**	**–**	**–**	**214**	**274**	**324**	**342**	**337**	**315**	**309**	**310**	**297**	**23**
Site Operating Costs														
Mining	$M	573	0	36	57	65	67	61	59	61	56	60	46	4
Processing Zn	$M	75	–	–	8	8	8	9	9	8	8	8	8	1
Processing Cu (tolling)	$M	12	–	–	0	4	4	1	1	0	0	2	–	–
General & Administration	$M	180	30	30	12	12	12	12	12	12	12	12	12	12
Total Operating Costs	**$M**	**840**	**30**	**66**	**77**	**88**	**91**	**83**	**80**	**81**	**77**	**82**	**66**	**17**
Operating Surplus / (Deficit)	**$M**	**1,904**	**-30**	**-66**	**137**	**187**	**233**	**259**	**257**	**233**	**231**	**228**	**231**	**6**
Capital Costs														
Rehabilitation	$M	111	100	11	–	–	–	–	–	–	–	–	–	–
Mining	$M	136	–	52	16	12	11	8	10	6	11	9	1	–
Process & Infrastructure	$M	38	–	32	–	1	1	1	1	1	1	1	–	–
Closure	$M	20	–	–	–	–	–	–	–	–	–	–	–	20
EPCM	$M	32	15	14	2	–	–	–	–	–	–	–	–	–
Owners	$M	32	15	14	2	–	–	–	–	–	–	–	–	–
Contingency	$M	63	30	29	5	–	–	–	–	–	–	–	–	–
Total Capital	**$M**	**431**	**160**	**152**	**25**	**13**	**11**	**9**	**11**	**7**	**12**	**10**	**1**	**20**
Cash Flow Before Tax	**$M**	**1,473**	**-190**	**-219**	**112**	**174**	**221**	**249**	**245**	**226**	**220**	**218**	**230**	**-14**
Income Tax	$M	396	–	–	–	–	–	55	74	68	66	65	69	–
Cash Flow After Tax	**$M**	**1,076**	**-190**	**-219**	**112**	**174**	**221**	**195**	**172**	**158**	**154**	**153**	**161**	**-14**
Change in Working Capital	$M	–	4	4	-25	-6	-6	-3	0	3	0	0	-0	29
Free Cash Flow After Tax	**$M**	**1,076**	**-187**	**-214**	**86**	**168**	**216**	**192**	**172**	**161**	**154**	**153**	**161**	**15**

22.3 Comparison to Other Projects

The Kipushi Project Mineral Resource Estimate, January 2016 includes Measured and Indicated Resources of 10.2 Mt at 34.89% 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 20 Zinc Projects by Contained Zinc

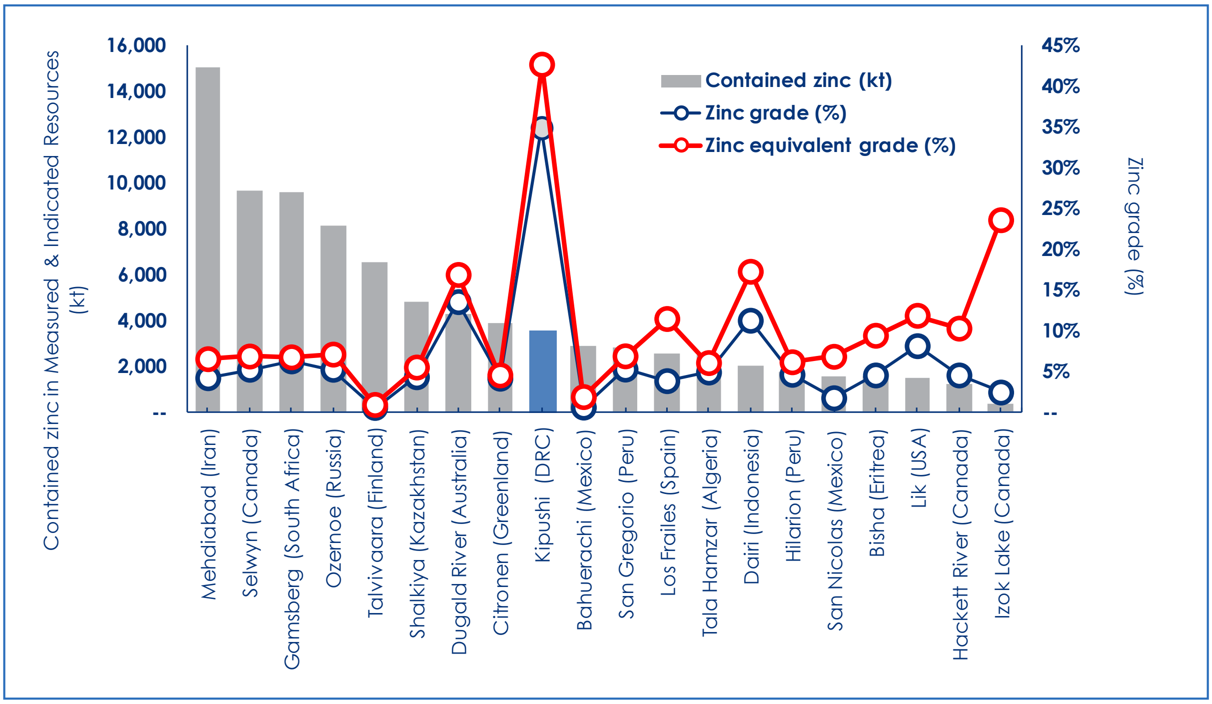


Figure by Wood Mackenzie, 2016.

Note: All tonnes and metal grades of individual metals used in the equivalency calculation of the above mentioned projects (except for Kipushi) are based on public disclosure and have been compiled by Wood Mackenzie. All metal grades have been converted by Wood Mackenzie to a zinc equivalent grade at price assumptions of US$1.01/lb Zn, US$2.86/lb Cu, US$0.91/lb Pb, US$12.37/lb Co, US$1,201/oz Au, US$17/oz Ag and US$2,000/kg Ge.

Life-of-mine average planned zinc concentrate production of 530,000 dry tpa, with a concentrate grade of 53% Zn, is expected to rank the Kipushi Zn-Cu Project, once in production, among the world's major zinc mines (Figure 22.6).

Figure 22.6 World's Major Zinc Mines (1), Showing Estimated Annual Zinc Production and Zinc Head Grade

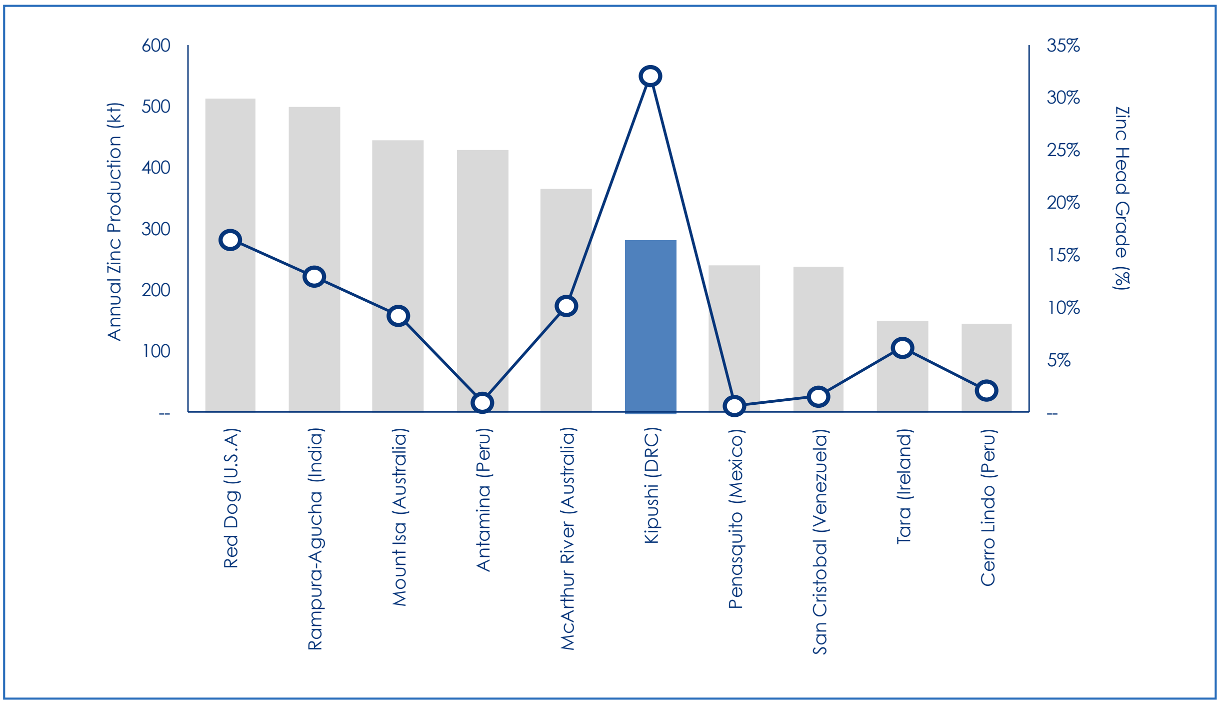


Figure by Wood Mackenzie, 2016.

(1) World's major zinc mines defined as the world's 10 largest zinc mines ranked by forecasted production by 2018

Note: Independent research by Wood Mackenzie compared the Kipushi Project's life-of-mine average annual zinc production and zinc head grade of 281,000 tonnes and 32%, respectively, against production and zinc head grade forecasts for 2018.

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.

Figure 22.7 Capital Intensity for Zinc Projects

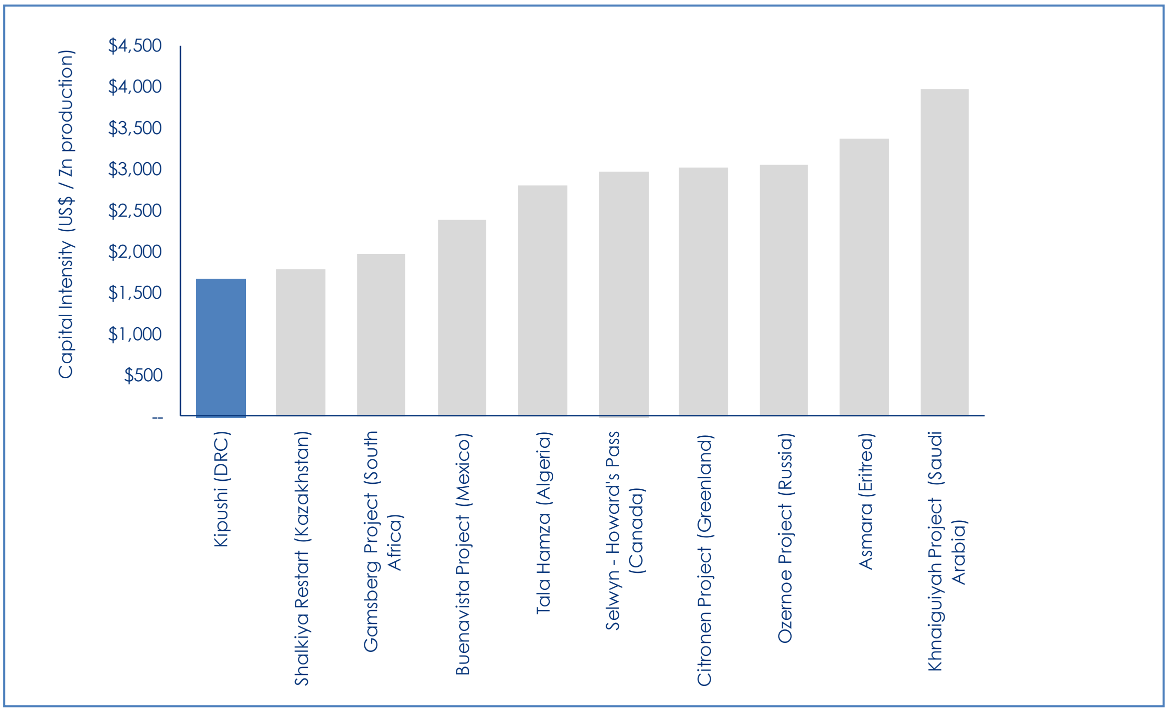


Figure by Wood Mackenzie, 2016.

Note: All comparable "probable" and "base case" projects as identified by Wood Mackenzie, based on public disclosure and information gathered in the process of routine research. The Kipushi 2016 PEA has not been reviewed by Wood Mackenzie.

Based on data from Wood MacKenzie, life-of-mine average cash cost of US$0.54/lb of zinc is expected to rank the Kipushi Zn-Cu Project, once in production, in the bottom quarter of the 2018 cash cost curve for zinc producers globally (Figure 22.8).

Figure 22.8 2018 Expected C1 Cash Costs

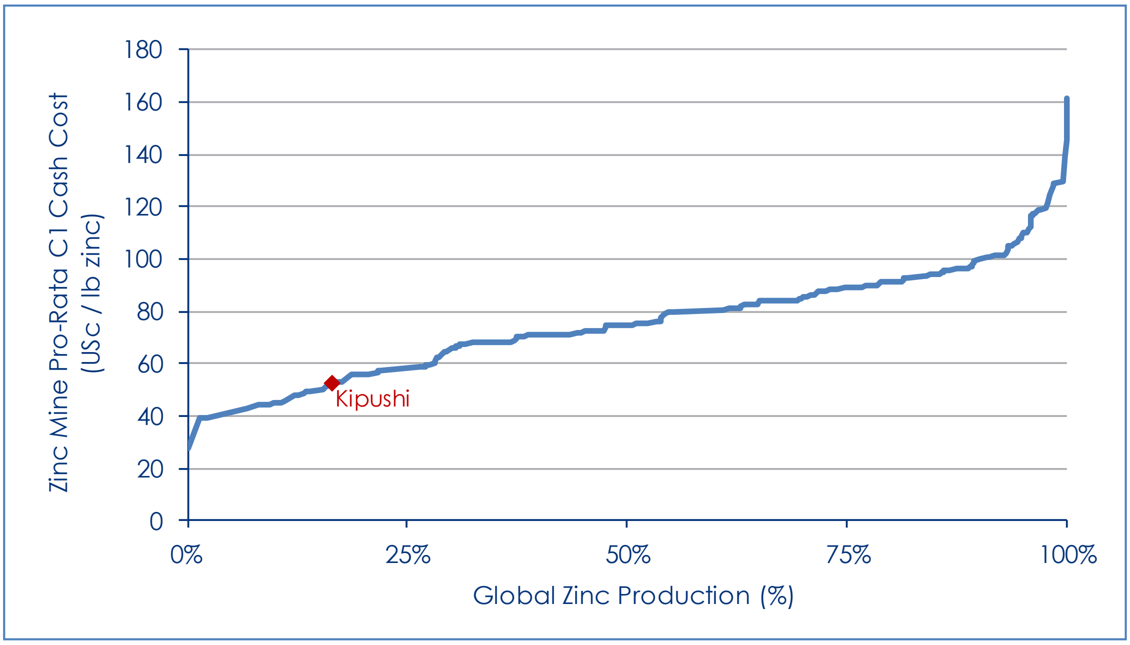


Figure by Wood Mackenzie, 2016.

Note: Represents C1 pro-rata cash costs which reflect the direct cash costs of producing paid metal incorporating mining, processing and offsite realization costs having made appropriate allowance for the co-product revenue streams. Based on public disclosure and information gathered in the process of routine research. The Kipushi 2016 PEA has not been reviewed by Wood Mackenzie.

23 ADJACENT PROPERTIES

This section not used.

24 OTHER RELEVANT DATA AND INFORMATION

This section not used.

25 INTERPRETATION AND CONCLUSIONS

The Kipushi 2016 PEA for the redevelopment of the Kipushi Mine is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorized as Mineral Reserves, and there is no certainty that the results will be realized. Mineral Resources do not have demonstrated economic viability and are not Mineral Reserves.

The Kipushi 2016 PEA has identified a positive business case and it is recommended that the Kipushi Zn-Cu Project is advanced to a pre-feasibility study level in order to increase the confidence of the estimates. There are a number of areas that need to be further examined and studied and arrangements that need to be put in place to advance the development of the Kipushi Zn-Cu Project. The key areas for further work are:

Resources

- Further updates and resource estimation.
- Additional drilling of the lower Big Zinc zone and possible extensions.
- Test drilling of the copper zones.
- Additional resource estimation of other elements.

Geotechnical

- Further geotechnical drilling and logging will be required in the next stage of the project to increase the confidence in geotechnical data.
- The direction of drilling in the next stage should be along strike to avoid an orientation bias, as the majority of drilling at this stage is in the dip direction of the various mineralised zones.
- Laboratory testing of the rock units to investigate the rock properties of all rock units.
- Underground mapping should be carried out to improve confidence in the joint orientations and rock mass classification.

Mining

- Complete shaft and underground rehabilitation work.
- Additional study work to define the declines, ventilation, and material handling pass systems.
- Detailed design and optimisation including geotechnical recommendations.
- Prepare detail material flow designs.
- Mine stope and sequencing optimisation, and geotechnical review.
- Material handling / ventilation review and refinement of refurbishment requirements.

Process

- Further metallurgical testwork, aligned to predicted head grades, including DMS testwork on variability samples over a range of zinc feed grades and locations and bulk sample and pilot programme using DMS and spirals to confirm the design criteria across a DMS / Spiral circuit.
- Basic engineering for DMS and associated bagging plant.
- Copper rich-zone testwork.
- Study of the production of other metal concentrates or pure metals in particular copper, lead, cadmium, germanium, and silver.
- Study of the potential production of zinc calcine, zinc metal, and acid.

Infrastructure

- Define the rail option development.
- Progress agreements for rail transport and engage with the rail contractor.
- Evaluate container shipping with shipping companies.
- Investigate permitting of Kipushi station for the rail yard plans.
- Investigate the track conditions from Kipushi to the main Lubumbashi line.
- Containerisation.
- Analyse detailed power requirements and negotiate with supplier.
- Site survey.

Marketing

- Investigate customer uptake for container transport.
- Investigate copper sales at mine gate opportunities.

Environmental and Social

- Complete the regulatory Environmental Impact Statement (EIS) and the Environmental Management Plan (EMPP).
- Identify other permitting requirements.
- Prepare detailed closure plan.

Project Financing

- Investigate financing options and sources.
- Review of capital and operating cost estimates as part of the pre-feasibility study.

26 RECOMMENDATIONS

26.1 Further Studies

The Kipushi 2016 PEA has identified a positive business case and it is recommended that the Kipushi Project is advanced to a pre-feasibility study level in order to increase the confidence of the estimates. There are a number of areas that need to be further examined and studied and arrangements that need to be put in place to advance the development of the Kipushi Zn-Cu Project.

The results of the Kipushi 2016 PEA suggest that further study should be undertaken. In particular, the investigation of logistics and transport, mining method and processing.

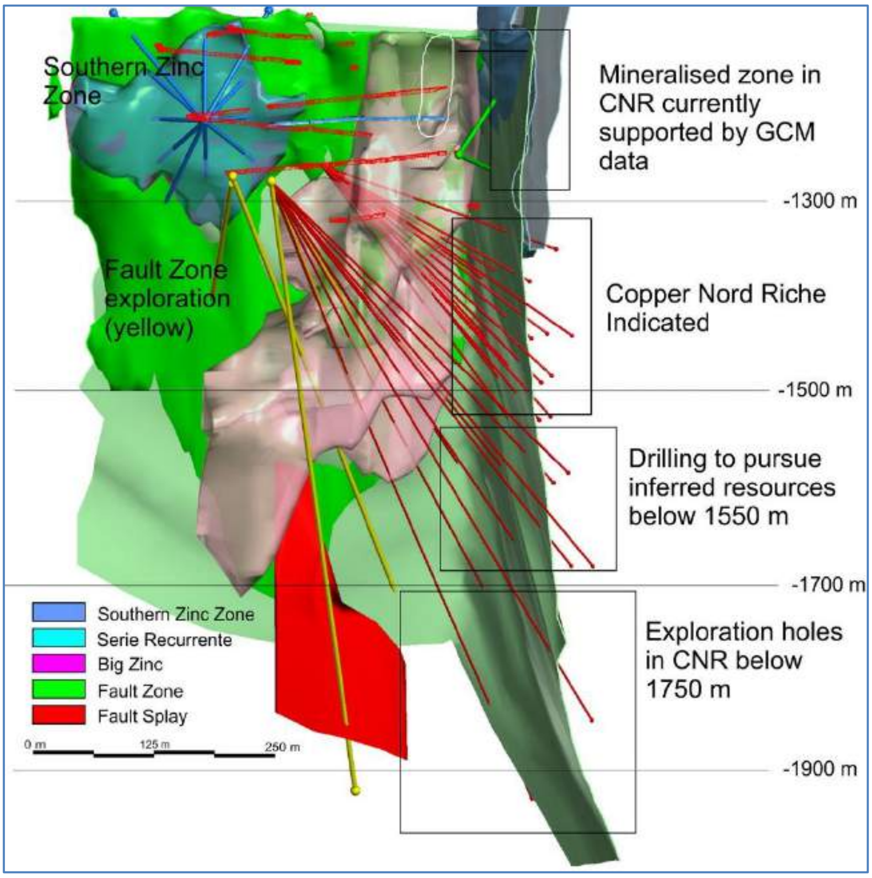
26.2 Geology and Resources

Approximated 16,500 m of drilling are recommended to aim to achieve both an Indicated Mineral Resource category on the Southern Zinc and Copper Nord Riche mineralized zones and to explore additional parts of the deposit that were not drilled during the 2014–2015 drilling campaign. Zones with the planned drilling are shown in Figure 26.1. A summary of the total metres is shown in Table 26.1.

Four holes are planned in the upper portion of the Copper Nord Riche zone to support previous Gécamines drilling and to attempt to bring this to an Indicated Mineral Resource category. Similarly, the Southern Zinc zone is not supported by Gécamines drilling and an additional 13 holes are recommended to attempt to achieve an Indicated Mineral Resource category. Further drilling is required to explore the Fault Zone and Copper Nord Riche at depth. The morphology of the deposit, together with the proximity of the supporting infrastructure to the steeply plunging mineralised zone, limit the options for deep pierce points within the Kipushi deposit.

The cost of the drilling programme is estimated at US$3.96 million. In the opinion of the MSA QPs (Michael Robertson and Jeremy Witley), the recommended work programme is considered appropriate and warranted in order to upgrade the Mineral Resource status of the Kipushi Zn-Cu Project.

Figure 26.1 Planned Drilling at Kipushi



Source: Ivanhoe Mines (2015)

Table 26.1 Planned Drilling by Zone

Planned drilling metres to achieved Mineral Resource class			
Mineralised Zone	**Indicated**	**Inferred**	**Exploration Drilling**
Copper Nord Riche (supporting Gécamines drilling)	704	–	–
Copper Nord Riche	4,589	4,301	2,390
Fault Zone	–	–	2,806
Southern Zinc	1,571	–	–

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