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IVANHOE MINES LTD.

Kipushi Project

Kipushi 2019 Resource Update

March 2019

Job No. 18005

IMPORTANT NOTICE

This notice is an integral component of the Kipushi 2019 Resource Update (Technical Report) and should be read in its entirety and must accompany every copy made of the Technical Report. The Technical Report has been prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects.

The Technical Report has been prepared for Ivanhoe Mines Limited (Ivanhoe) by OreWin Pty Ltd (OreWin), The MSA Group (Pty) Ltd (MSA), SRK Consulting (South Africa) (Pty) Ltd (SRK) and MDM (Technical) Africa Pty Ltd (MDM) (a division of Wood PLC). The Technical Report is based on information and data supplied to OreWin and MSA by Ivanhoe and other parties and where necessary the authors have assumed that the supplied data and information are accurate and complete.

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

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

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

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

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Title Page

Project Name:	Kipushi Project
Title:	Kipushi 2019 Resource Update
Location:	Haut-Katanga Province
	Democratic Republic of the Congo
Effective Date of Technical Report:	28 March 2019
Effective Date of Mineral Resources:	14 June 2018
Effective Date of Drilling Database:	24 April 2018
Effective Date of Mineral Reserves:	12 December 2017

Qualified Persons:

- Bernard Peters, B. Eng. (Mining), FAusIMM (201743), employed by OreWin as Technical Director - Mining was responsible for: Sections 1.1, 1.3, 1.4, 1.10, 1.11, 1.11.1, 1.11.2, 1.11.7, 1.11.8, 1.11.9, 1.12, 1.13.3. 1.13.6 to 1.13.8; Sections 2 to 5; Section 15; Sections 16, 16.2 to 16.16; Section 19; Section 20; Sections 21, 21.1 to 21.3, 21.5 to 21.6; Sections 22 to 24; Sections 25.3, 25.6 to 25.8; Section 26.3; Section 27.
- Michael Robertson, BSc Eng (Mining Geology), MSc (Structural Geology), Pr.Sci.Nat SACNASP, FGSSA, MSEG, MSAIMM, employed by The MSA Group (Pty) Ltd as a Principal Consulting Geologist was responsible for: Sections 1.5 to 1.8.2, 1.13, 1.13.1; Sections 2 to 3; Sections 6 to 12; Sections 25, 25.1; Section 26.2; Section 27.
- Jeremy Witley, BSc Hons (Mining Geology), MSc (Eng), Pr.Sci.Nat SACNASP, FGSSA, employed by The MSA Group (Pty) Ltd as a Principal Resource Consultant was responsible for: Sections 1.2, 1.13.1; Sections 2 to 3; Section 14; Section 25.1; Section 26.1; Section 27.
- William Joughin, FSAIMM (55634), employed by SRK Consulting (South Africa) (Pty) Ltd as Principal Consultant, was responsible for: Section 1.13.2; Sections 2; Sections 16.1; Section 25.2.
- Dean David, FAusIMM (CP) (102351), B App Sc (Metallurgy), employed by Wood (mining and Metals Australia West) as Technical Director – Process, was responsible for: Sections 1.9, 1.11.3 to 1.11.6, 1.13.4, 1.13.5; Section 13; Section 17; Section 18; Section 21.4; Sections 25.4, 25.5; Section 26.4; Section 27.

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Signature Page

Project Name:	Kipushi Project
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Location:	Haut-Katanga Province
	Democratic Republic of the Congo
Effective Date of Technical Report:	28 March 2019

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1 SUMMARY

1.1 Introduction

The Kipushi 2019 Resource Update has been prepared for Ivanhoe Mines Ltd. (Ivanhoe), to update the Mineral Resources for the Kipushi Project (the Project) located in the Democratic Republic of Congo (DRC). The Kipushi 2019 Resource Update is an independent NI 43-101 Technical Report (the Report) prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for Ivanhoe.

The Project is located adjacent to the town of Kipushi in the south-western part of the Haut-Katanga Province in the DRC, adjacent to the border with Zambia. Kipushi town is situated approximately 30 km south-west of Lubumbashi, the capital of Haut-Katanga Province. Kipushi Holding Limited (a 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 Project. Ivanhoe and Gécamines respectively own 68% and 32% of the Kipushi Project through Kipushi Corporation SA (KICO), the mining rights holder of the Kipushi Project.

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

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

The previous Technical Report was the Kipushi 2017 Prefeasibility Study on the Kipushi Project. Ivanhoe has undertaken further mineral resource studies following the Kipushi 2017 PFS that has formed the basis of the Kipushi 2019 Resource Update, which summarises the current Ivanhoe development strategy for the Kipushi Project. The Kipushi 2019 Resource Update provides an update of the Kipushi Project Mineral Resource, with the Mineral Reserve from the Kipushi 2017 PFS remaining the same.

Other than the addition of information relevant to the reporting of the Kipushi Resource, the remainder of this is report has not been changed from the Kipushi 2017 PFS and remains the most current study work available. KICO are preparing a feasibility study on Kipushi, with results to be published once the study work has been completed. The Kipushi feasibility study work is currently incomplete and has not determined any results that require material changes to the Kipushi 2017 PFS. The Kipushi 2019 Resource Update should be read in this context.

1.2 Mineral Resource Estimates

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

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

In total, 106 Gécamines holes and 134 KICO holes were used for the grade estimate. The cut-off date for data included in this estimate is 26 April 2018, there being no additional drilling data collected since then. The Mineral Resource was estimated using The Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Best Practice Guidelines and is reported in accordance with the 2014 CIM Definition Standards, which have been incorporated by reference into National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101).

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

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

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

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

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

Note:

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

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

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

Note:

1. All tabulated data has been rounded and as a result minor computational errors may occur.
2. Mineral Resources which are not Mineral Reserves have no demonstrated economic viability.
3. The Mineral Resource is reported as the total in-situ Mineral Resource and on a 100% project basis, exclusive of Mineral Reserves. Ivanhoe holds an indirect 68% interest in the Project.
4. Metal quantities are reported in multiples of Troy Ounces or Avoirdupois Pounds.
5. The cut-off grade calculation was based on the following assumptions: copper price of US$3.00/lb, mining cost of US$50/tonne, processing cost of US$10/tonne, G&A and holding cost of US$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 1.3, and the Inferred Mineral Resource in Table 1.4.

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

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

Note:

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

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

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

Note:

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

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 1.5, and the Inferred Mineral Resource in Table 1.6.

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

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

Note:

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

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

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

Note:

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

Mineral Resource estimates were completed below the 1,150 mRL on the Big Zinc, Southern Zinc, Fault Zone and Série Récurrente, 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 30 years' experience in base and precious metals exploration and mining as well as Mineral Resource evaluation and reporting. He is a Principal Mineral Resource Consultant for The MSA Group (an independent consulting company), is registered with the South African Council for Natural Scientific Professions (SACNASP) and is a Fellow of the Geological Society of South Africa (GSSA). Mr. Witley has the appropriate relevant qualifications and experience to be considered a "Qualified Person" for the style and type of mineralisation and activity being undertaken as defined in National Instrument 43-101 Standards of Disclosure of Mineral Projects.

1.3 Property Description and Ownership

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

The Kipushi mine is a past-producing, high-grade underground zinc–copper mine in the Central African Copperbelt, which operated from 1924 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.

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 Project. Ivanhoe and Gécamines respectively own 68% and 32% of the Kipushi Project through Kipushi Corporation SA (KICO), the mining rights holder of the Kipushi Project.

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

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

1.4 Mineral and Surface Rights, Royalties, and Agreements

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

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

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

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

The boundaries of Exploitation Permit No. 12434, indicated in the Exploitation Certificate, cross the international border, as do some of the 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 for KICO to mine stops at the international border, and any part of the exploitation permit area extending beyond the DRC borders are excluded from the exploitation permit.

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

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. It is contained in the Katangan basin, an intracratonic rift that records onset of growth at ~840 Ma and inversion at ~535 Ma (Selley et al., 2018). The succession is divided into three regionally mappable groups, which from oldest to youngest are named the Roan, Nguba, and Kundelungu Groups.

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

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

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

1.6 Exploration

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

1.7 Drilling Programmes

1.7.1 Gécamines Drilling

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

1.7.2 KICO Drilling

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

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

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

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

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

1.8 Sample Preparation and Analysis

1.8.1 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 QA/QC protocols implemented for the Gécamines samples and therefore the Gécamines sample assays were considered to be less reliable than the KICO sample assays.

1.8.1.1 Resampling Programme

A comprehensive resampling programme was undertaken on historical Gécamines drill core from the Big Zinc and Fault Zone below 1,270 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 between March 2014 and May 2015 with the objective of verifying historical Gécamines results. It was concluded that the results of the drill core resampling programme confirm that the assay values reported by Gécamines are reasonable and can be replicated within a reasonable level of error by international accredited laboratories under strict QA/QC control.

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

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

1.8.1.2 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:

Density = 2.85 + 0.039 x Cu% + 0.0252 x Pb% + 0.0171 x 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.

1.8.2 KICO Sample Preparation and Analysis

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

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

Sample preparation was completed by staff from KICO and its affiliated companies at its own internal containerised laboratories at Kolwezi and Kamoa-Kakula. Between 1 June and 31 December 2014, samples were prepared at the Kolwezi sample preparation laboratory by staff from the company's exploration division. After 1 January 2015, samples were prepared at Kamoa-Kakula by staff from that project. 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.
- A 90 g sample for office archives.

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

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

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

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

- Zn, Cu, and S assays by SPF with ICP-OES finish.
- Pb, Ag, As, Cd, Co, Ge, Re, Ni, Mo, V, and U assays by peroxide fusion with ICP-MS finish.
- Ag and Hg by Aqua Regia digest with ICP-MS finish.
- Au, Pt, and Pd by 10 g (due to inherent high 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 (QA/QC) programme was maintained by KICO throughout the underground drilling campaign comprising drillholes KPU001 to KPU156. The QA/QC programme was established to monitor the quality of data for geological modelling and Mineral Resource estimation. All KICO data from the project are stored in an MS Access database. QA/QC data were exported from the MS Access database into software applications for creating monitoring charts and comparison charts.

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

1.9 Metallurgical Testwork Summary

Metallurgical testwork program were completed on drill core samples of known Kipushi mineralisation between 2013 and 2017 for the various project redevelopment study phases. These investigations were focused on metallurgical characterisation and flowsheet development for the processing of material from the Big Zinc Area.

In 2013, scoping testwork on 60 kg Kipushi quarter-core was analysed and scoping testwork completed at Mintek, South Africa. The scoping testwork included mineralogy, comminution and flotation tests. 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, predominantly sphalerite, 65.9%, followed by pyrite, 24%, with galena and chalcopyrite present in minor quantities. The major gangue minerals were silica and carbonaceous minerals. The sphalerite and galena are coarse grained, with grains up to 1 mm and 0.5 mm respectively. Chalcopyrite showed relatively fine grains, less than 0.04 mm.

Comminution testing showed this 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.

In 2015, approximately 400 kg of half core material was selected for the Kipushi 2016 PEA testwork. Mineralogy and gravity separation testwork was completed by Mintek, South Africa, and the results used as a basis of design for the Kipushi 2016 PEA. Six drillholes intercepting the Big Zinc 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 including, but not limited to, Massive Brown Sphalerite (MSB), Massive Sulphide Mixed (MSM), and Dolomite (SDO). The Kipushi 2016 PEA composite sample head analysis was 40% Zn, 1.45% Pb, 0.3% Cu, 25% S, and 6% Fe.

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 the by HLS method used.

The three crush sizes evaluated were –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% Zn. While the shaking table achieved 58% recovery at a concentrate grade of approximately 56% zinc. The difference in overall circuit performance of the three crush sizes is the mass percentage reporting to the –1 mm fine fraction processed through the less-efficient shaking tables which made 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 2016, approximately 900 kg of half core from eight drillholes intercepting the Big Zinc were selected and intervals composited for variability and flowsheet development testwork program ahead of the Kipushi 2017 PFS. About ten composites were constituted for variability tests using the physical separation circuit developed during the Kipushi 2016 PEA. A PFS development composite was also constituted for flowsheet development and optimisation testwork program. The PFS development composite intercepts were sampled from hole numbers: KPU001, KPU042, KPU085, and KPU086. The drill cores for the PFS composite sample, was selected to represent all mineralisation types in the Big Zinc including, but not limited to: Massive Brown Sphalerite (MSB); Massive Sulphide Mixed (MSM); and, Dolomite (SDO). Assayed intervals from the resource drill core, were used to derive a composite sample that had a similar feed composition to the LOM average head grade of 32% zinc as presented in Table 1.7.

Table 1.7 Kipushi Composite Sample Head Analysis Results

Element	Zn (%)	Pb (%)	Fe (%)	Ca (%)	Si (%)	Cu (%)	Mg (%)	S (%)
Average head assay	32.73	0.72	6.79	7.11	0.88	0.42	4.3	24.53

Mineralogical investigations conducted on the 2016 PFS development composite head sample confirmed that the Big Zinc is predominately sphalerite (49%), with chalcopyrite (1%) and galena (1%) present as minor constituents, the gangue minerals in order of abundance: dolomite (31%); pyrite (14%); quartz (2%). Grainsize analysis showed that sphalerite is coarse grained with an average grain size of 105 µm, while galena and chalcopyrite are finer with an average grain size <60 µm.

Gravity separation tests (Heavy liquid separation (HLS) and shaking table) tests were conducted on variability samples and the PFS composite sample, as per the Kipushi 2016 PEA flowsheet. Crushed material (-20 mm +1 mm) was subjected to HLS testwork, whilst crusher fines (–1 mm to +38 µm) was tested on bench scale shaking tables (shaking tables provide a laboratory scale simulation of a commercial spiral plant).

The PFS composite sample achieved 99% recovery at a concentrate grade of 49% zinc; whilst the shaking table achieved 77% recovery also at a concentrate grade of approximately 49% zinc. Gravity separation tests achieved overall high recovery >95% for all composites tested, however concentrate zinc grade was variable between 30 and 53% zinc depending on the base metal sulphides content of various feed samples. The old circuit showed that although the DMS plant was highly effective in rejecting dolomite, with limited loss in zinc, other heavy sulphide minerals associated with copper; lead; and iron, reported to the concentrate and consequently diluted the concentrate zinc grade below saleable concentrate specification.

Variability simulations on the basis of the Kipushi 2016 PEA flowsheet were undertaken in METSIM® using the expected range of ROM mineralogical compositions over the LOM. These simulations further confirmed that the Kipushi 2016 PEA circuit could not consistently produce zinc concentrate that meets required specification because other heavy sulphide minerals associated with copper lead and iron also reported to concentrate. Furthermore, input from KICO suggested that a fine (µm), rather than coarse (mm) concentrate was required by the custom smelters.

On the above basis, KICO undertook further testwork that incorporated a milling and flotation circuit, specifically to ensure a saleable zinc concentrate (P_{100} <500 µm and >52% Zn).

Two flowsheet options were tested, the results of which formed the basis for a conceptual techno-economic trade-off study conducted by MDM with the objective of selecting the optimal process flowsheet to be further developed to the level of detail required to support the PFS.

The two options evaluated are listed below.

- Option I - Full stream ROM milling (P_{80} = 106 µm) followed by differential flotation.
- Option II - DMS pre-concentration followed by the milling (P_{80} = 106 µm) and differential flotation of the DMS concentrate and the crusher circuit's fine fraction (-1 mm).

The differential flotation circuit tests were conducted using the flotation feed material as specified above. In the differential float, a copper/lead concentrate was first produced, followed by zinc flotation and pyrite depression in the subsequent flotation stage. The zinc rougher tails and the copper/lead concentrate were discarded as final tails. Duplicates tests results for Option 1 and Option 2, produced overall zinc recoveries of 94% and 90% at a concentrate grade of 54% Zn and 60% Zn, respectively.

The results of a high-level techno-economic analysis favoured Option II, which was chosen as the optimal circuit as it reduced mass pull; transport costs; tailings storage requirements and provides DMS tails required for the mining backfill.

On the above basis, Option 2 was developed to the level of engineering and technical detail, required to support a PFS. The testwork undertaken thus far, suggests that for the average weighted LOM zinc head grade will produce a zinc concentrate grading 59% zinc at an overall (steady state) recovery of 89.6%.

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 pre-concentrated through a Dense Media Separation at a density of 3.1 g/cm^3 and the screen undersize material (–1 mm) will be combined with the DMS sinks to milling and flotation circuit.

1.10 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 2018, KICO employed approximately 400 people.

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

1.11 Kipushi 2017 Prefeasibility Study

The Kipushi 2019 Resource Update includes restatement of the Kipushi 2017 Prefeasibility Study which includes the Kipushi Mineral Reserve. The Mineral Reserve in the Kipushi 2017 PFS remains valid. Further study work is currently incomplete and has not determined any results that require material changes to the Kipushi 2017 PFS.

Underground mining of the Big Zinc is planned to be undertaken using a Sublevel Open Stoping (SLOS) method. The mine production is expected to be 0.8 Mtpa. Underground tonnes are anticipated to be mined, crushed in underground facilities and hoisted to the surface via Shaft 5. The crushed material is expected to be pre-concentrated in a dense media separation (DMS) plant, followed by milling and flotation to produce saleable concentrate.

Life-of-mine average annual planned zinc concentrate production is anticipated to be 381 ktpa, with a concentrate grade of 59% Zn. Total zinc production is anticipated to be 8.6 Mt ore at 32.14% Zn over a period of eleven years to produce 2,472 kt of zinc metal in concentrate.

Concentrate is planned to be transported by rail directly from a new loading terminal at the Kipushi Mine to either the port of Durban or Richards Bay in South Africa, from where it would be shipped by sea to customers.

The estimates of cash flows have been prepared on a real basis as at 1 January 2018 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 US$2,425/t Zn. This price is 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 $683 M.
- After-tax internal rate of return (IRR) is 35.3%.
- After-tax project payback period is 2.24 years.

The key results of the Kipushi 2017 PFS are summarised in Table 1.8.

Table 1.8 Kipushi 2017 PFS Results Summary

Item	Unit	Total
Zinc Ore Processed		
Quantity Zinc Ore Treated	kt	8,581
Zinc Feed grade	%	32.14
Zinc Concentrate Recovery	%	89.61
Zinc Concentrate Produced	kt (dry)	4,196
Zinc Concentrate Grade	%	58.91
Metal Produced		
Zinc	Mlb	5,449
Key Financial Results		
Pre-Production Capital	US$M	337
Mine Site Cash Cost	US$/lb Payable Zn	0.14
Realisation	US$/lb Payable Zn	0.35
Total Cash Costs	US$/lb Payable Zn	0.48
Site Operating Costs	US$/t milled	87.77

The key economic assumptions for the analyses are shown in Table 1.9.

Table 1.9 Metal Prices and Terms

Parameter	Unit	Financial Analysis Assumption
Zinc Price	US$/lb	1.10
Zinc Treatment Charge	$/t concentrate	170.00

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 1.10.

Table 1.10 Financial Results

	Discount Rate	Before Taxation	After Taxation
Net Present Value (US$M)	Undiscounted	1,944	1,435
	5.0%	1,239	900
	8.0%	953	683
	10.0%	743	517
	12.0%	628	431
	15.0%	487	325
	18.0%	401	262
	20.0%	335	213
Internal Rate of Return	–	41.7%	35.3%
Project Payback Period (Years)	–	1.9	2.2

Table 1.11 After Tax NPV_8 Sensitivity to Zinc Price and Discount Rates

Discount Rate (%)	Zinc (US$/lb)								
	0.80	0.90	1.00	1.10	1.20	1.40	1.50	1.70	2.00
Undiscounted	516	823	1,129	1,435	1,742	2,355	2,661	3,274	4,193
5%	254	472	687	900	1,111	1,533	1,744	2,165	2,796
8%	150	331	508	683	855	1,199	1,370	1,713	2,226
10%	96	257	414	568	719	1,021	1,172	1,473	1,923
12%	51	195	335	471	605	872	1,005	1,271	1,668
15%	-2	121	239	354	467	691	802	1,025	1,357
18%	-42	63	164	262	358	548	642	831	1,112
20%	-64	32	124	213	299	470	555	724	977

Note: Table shows NPV_8 $M.

1.11.1 Mineral Reserves

The Kipushi 2017 PFS Mineral Reserve has been estimated by Qualified Person Bernard Peters, Technical Director – Mining, OreWin Pty. Ltd., using the 2014 CIM Definition Standards. The Mineral Reserve is based on the January 2016 Mineral Resource. The effective date of the Mineral Reserve statement is 12 December 2017. Table 1.12 shows the total Proved and Probable Mineral Reserve of Kipushi.

Table 1.12 Kipushi Proved and Probable Reserve – Tonnage and Grades

Category	Tonnage (Mt)	Zn (%)	Contained Zn (kt)
Proved	3.10	35.41	1,098
Probable	5.48	30.29	1,660
Total	**8.58**	**32.14**	**2,758**

1. Effective date of the Mineral Reserves is 12 December 2017.
2. Net Smelter Return (NSR) is used to define the Mineral Reserve cut-offs, therefore cut-off is denominated in US$/t. By definition the cut-off is the point at which the costs are equal to the NSR. An elevated cut-off grade of US$135/t NSR (14.03% Zn) was used to define the mining shapes. The marginal cut-off grade has been calculated to be US$51/t NSR (3.43% Zn).
3. Mineral Reserves are based on a zinc price of $1.01/b Zn and a treatment charge of $200/t concentrate.
4. Economic analysis to demonstrate the Kipushi 2017 PFS Mineral Reserve has used a zinc price of $1.10/lb Zn and a treatment charge of $170/t concentrate.
5. Only Measured Mineral Resources were used to report Proven Mineral Reserves and only Indicated Mineral Resources were used to report Probable Mineral Reserves.
6. Mineral Reserves reported above were not additive to the Mineral Resources and are quoted on a 100% project basis.
7. Totals may not match due to rounding.

1.11.2 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 in the footwall of the fault that is approximately east–west striking and steeply north dipping.

KICO has a significant amount of underground infrastructure at the Kipushi Project, including a series of vertical mine shafts, with associated 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, 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

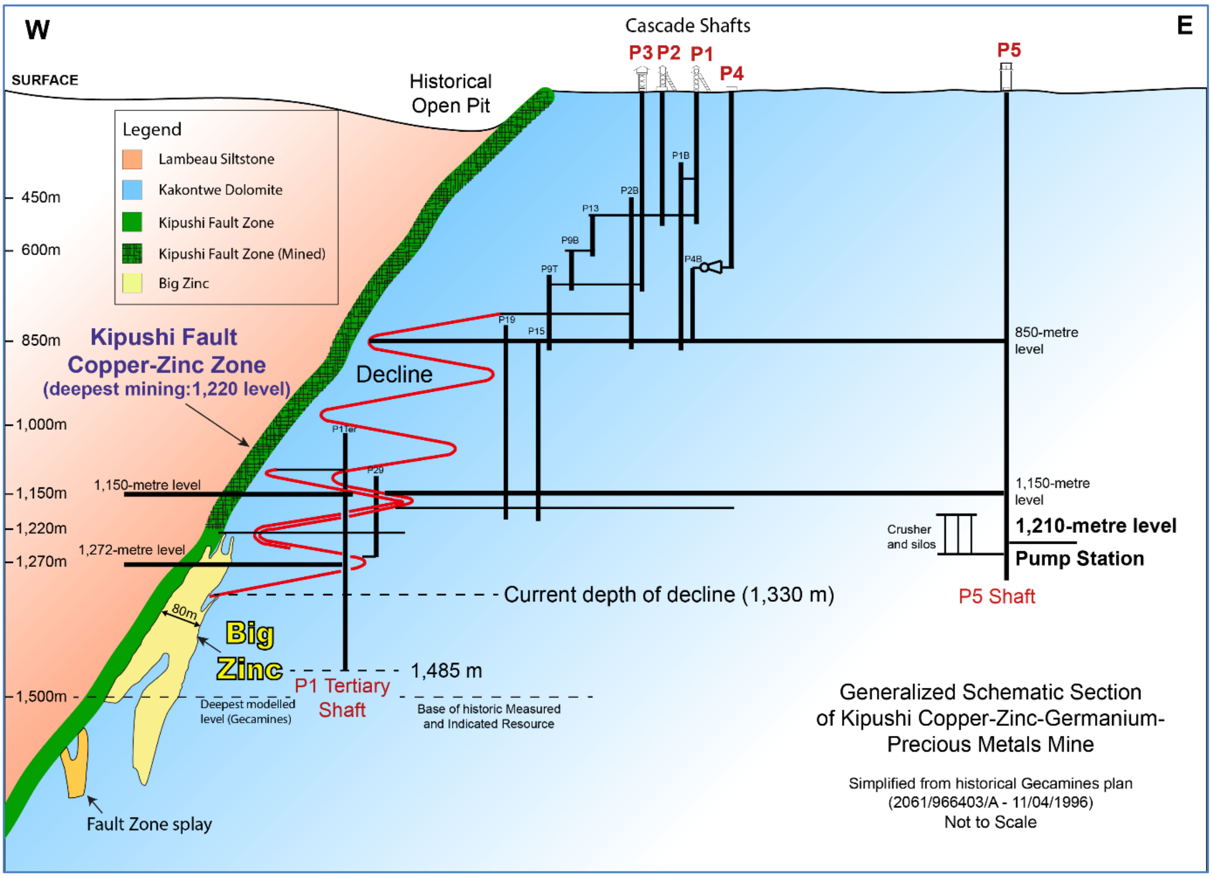


Figure by Ivanhoe, 2016.

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

Mining is planned to be a combination of longitudinal SLOS and Pillar Retreat methods. The Big Zinc mining method is expected to be longitudinal SLOS with mined stopes backfilled with CRF after stoping. The sill pillars are expected to be mined using the Pillar Retreat mining method once the adjacent stopes are backfilled.

The Big Zinc is expected to be accessed via the existing decline and without significant new development. The decline is planned to be developed from the existing level at approximately 1,330 mRL to the bottom stoping level at 1,590 mRL. The zinc stoping is expected to be carried out between 1,207 mRL and 1,590 mRL, and the uppermost stoping level on the Big Zinc is planned to be the 1,245 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 a single parallel drive in each stope. The sill pillar height is planned to be 15 m. Stopes are planned to be mined 60 m along strike and then filled with CRF. Remote capable loaders are expected to be used for loading the broken rock beyond the stope brow. The existing and planned development and stoping is shown in Figure 1.2.

Figure 1.2 Planned Kipushi 2017 PFS and Existing Development

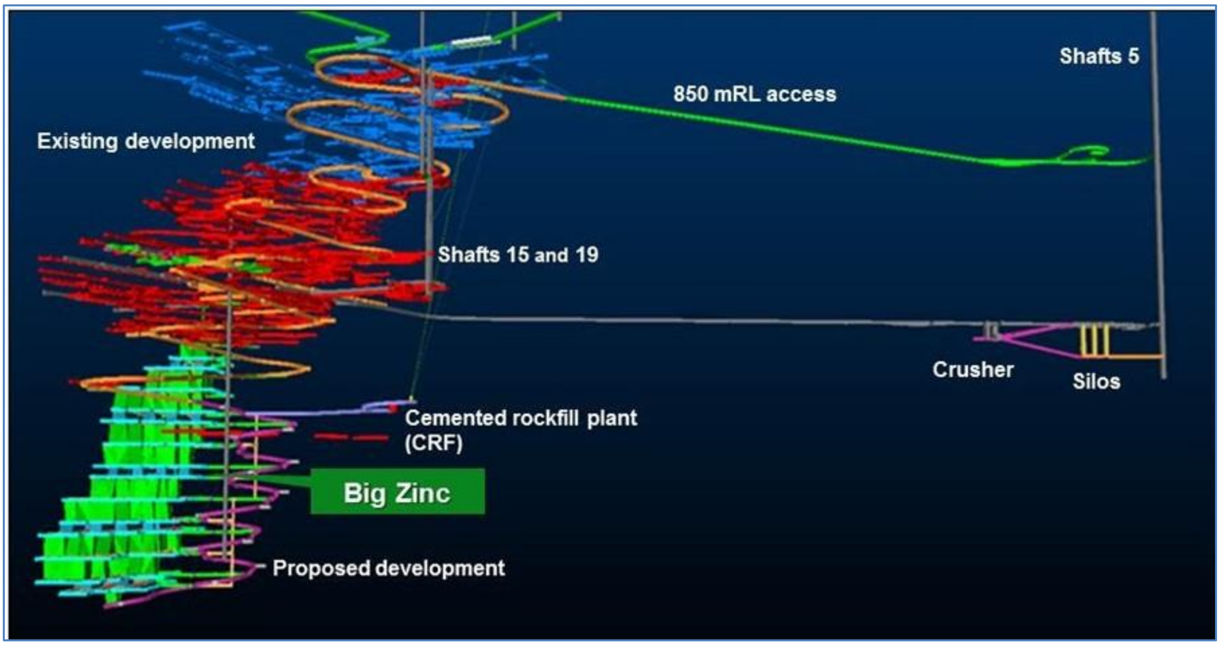


Figure by OreWin, 2017.

1.11.3 Processing Plant

The process plant as currently proposed has a name plate capacity of 800 ktpa, a nominal design Life-of-Mine (LOM) head grade of 32.14% Zn, a production life of 11 years and an average LOM zinc recovery of 89.6%. The installed power for the process plant is 4.6 MW. The process plant consists of two-stage crushing and screening, dense media separation, ball mill grinding, and differential flotation circuit, thickening and filtration, producing a saleable zinc concentrate which is sold.

Ore and waste from the Big Zinc is crushed underground to a product size of 100% passing 200 mm and hoisted to surface using Shaft 5. Both crushed ore and development waste will be intermittently (and separately) hoisted to surface, depositing into a single bin on surface, within the Shaft 5 headframe. Material is reclaimed from said bin via a vibrating feeder, which ultimately transfers to a single 900 m overland conveyor connecting Shaft 5, to the main mine area at the Old Kipushi Concentrator (OKC). The overland conveyor discharges the material into a crusher feed bin. Material is reclaimed through a feeder into a two-stage surface crushing plant. This plant consisting of two crushers and a double deck screen, ensures a -20+1 mm Dense Media Separation (DMS) plant feed product and minimal fines Screen fines (-1 mm) is combined with water and pumped to the mill discharge sump.

The screened -20+1 mm material will be subjected to the Dense Media Separation (DMS) at a density cut point of 3.1 t/ m^3 using atomised ferrosilicon as "medium" to separate the dense sphalerite and other minerals from the predominantly dolomitic waste. DMS concentrate is sent to the milling section whilst floats being dolomite is sent to the waste handling area.

The DMS concentrate and crusher fines are milled in a closed-circuit variable speed single stage ball mill, with cyclone classification to produce material of 80% passing 106 µm. The mill is fed at a controlled rate, with steel balls added manually onto the mill feed conveyor. The cyclone overflow gravitates to the flotation circuit at a solids density of 30%.

The milled slurry feeds a two-stage selective floatation circuit which selectively removes copper and lead for disposal and then floats a zinc concentrate. Mill slurry will be conditioned with reagents for copper and lead rougher flotation and the tails will again be conditioned with reagents suitable for zinc flotation. Zinc flotation concentrate will be thickened, filtered and bagged for loading onto train wagons ready for despatch to the market. The Cu/Pb concentrate is combined with zinc float tails, thickened and pumped to a new tailings storage facility. The DMS discard is stockpiled and used for cemented rock fill.

Life-of-mine average annual planned zinc concentrate production is anticipated to be 381 ktpa, with a concentrate grade of 59% Zn. Total zinc production is anticipated to be 8.6 Mt ore at 32.14% Zn over a period of eleven years to produce 2,472 kt of zinc metal in concentrate.

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

Figure 1.3 Kipushi Concentrator Plant Block Flow Diagram

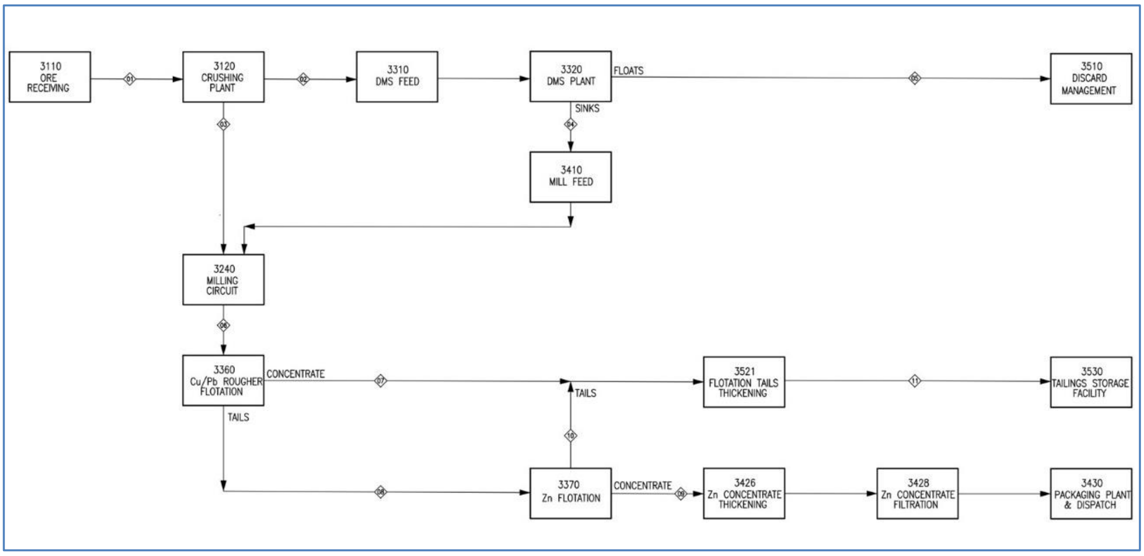


Figure by MDM, 2017.

1.11.4 Water Management

Underground water is planned for use as process water in the new process plant. Flotation tailings will be deposited in a new tailings storage facility (TSF) located south of the process plant as shown in Figure 1.4. In the proposed scheme, the return from the TSF is first neutralised and blended with the excess underground water before discharging to the Kipushi river via the north cut-off channel. A neutralisation plant has been included in the PFS as the geochemical analysis undertaken on the basis of available data indicated possible acidity of the TSF return water that, even after blending with underground water, falls outside DRC prescribed discharge limits.

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

Figure 1.4 Water Management Block Flow Diagram

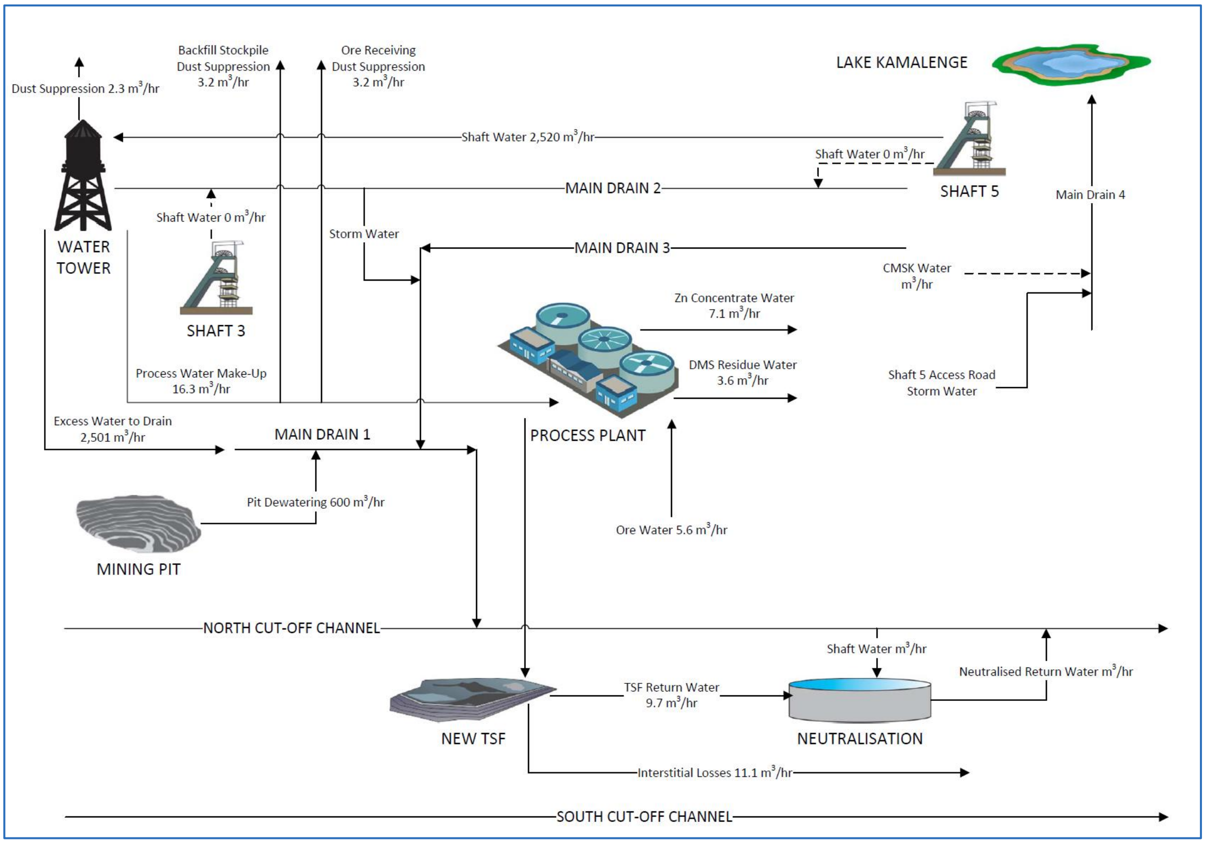


Figure by KICO, 2018.

1.11.5 Tailings Management and Disposal

The tailings storage facility (TSF) will store approximately 2 Mt of waste from the flotation plant. The tails stream comprises primarily of; chalcopyrite (Cu), galena (Pb), and pyrite (Fe), and some residual dolomite that was not recovered in the DMS plant.

Several sites were provisionally identified as potential sites for location of the TSF as shown in Figure 1.5. A ranking matrix identified Site 4 as the most optimal location for the TSF.

The key design features of the TSF are as follows:

- The TSF will be constructed as a full impoundment dam with a compacted earth wall.
- A liner system, including a double layer of 1500 micron HDPE geomembrane with associated leakage detection, leachate collection system and cushioning layers.
- An elevated toe filter drain and associated toe drain outlets and collection pipeline.
- Stormwater diversion/run-off trenches to divert rainfall run-off away from the facility.
- Phased construction, with an initial phase of 8.4 m high compacted earth starter impoundment yielding 2.5 years of storage capacity. Thereafter the construction of the impoundment walls has been phased, such that the impoundment crest elevation is at least two metres ahead of the tailings to allow for sufficient freeboard.

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

1.11.6 Infrastructure

The property hosts surface mining and processing infrastructure, a mineral processing/beneficiation plant, offices, workshops, stores, and connection to the national power grid. All of the surface infrastructure is owned by Gécamines, and is either ceded or leased to KICO. The overall proposed site layout is shown in Figure 1.6.

Key aspects of the project infrastructure are:

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

A combination of:

- Old (refurbished) and new facilities including:
  - General office, technical buildings and structures.
  - Mine services buildings (change rooms, mess, kitchen, laundry).
  - Workshops, stores and construction laydown areas.
  - General electrical buildings.
  - Security and emergency services buildings.

Figure 1.6 Overall Proposed Site Layout

Figure by Ivanhoe, 2017.

1.11.7 Concentrate Transport and Logistics

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

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

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 Munama and through to the Zambian boarder at Ndola.

On October 30, 2017, Ivanhoe Mines and the DRC's state-owned railway company, Société Nationale des Chemins de Fer du Congo (SNCC), signed a MOU to rebuild 34 kilometres of track to connect the Kipushi Mine with the DRC national railway at Munama, south of the mining capital of Lubumbashi.

Under the terms of the MOU, Ivanhoe has appointed R&H Rail (Pty) Ltd. to conduct a front-end engineering design study to assess the scope and cost of rebuilding the spur line from the Kipushi Mine to the main Lubumbashi-Sakania railway at Munama. The study also covers development of a preliminary operational plan. The study has begun and construction on the Kipushi-Munama spur line could start in late 2018. Ivanhoe will finance the estimated US$32 million (plus contingency) capital cost for the rebuilding, which is included within the overall Kipushi 2017 PFS capital cost.

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 North–South Rail Corridor to Durban via Zimbabwe is fully operational and has significant excess capacity.

For the direct rail option, the development of a rail loading facility at the mine and the rebuilding of the 34 km rail track between Kipushi and Munama, where it links up with the existing North–South Corridor, will be required. It is estimated that the rebuilding of the Kipushi to Munama railway line will take 23 months. Trains operated by SNCC can then be brought to the mine for loading and customs clearing can be done at the mine, before railing to the export ocean port, shown in Figure 1.7.

The existing Kipushi Station will 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.

The rail operator would need to source the fleet of rolling stock and establish a dedicated pool of wagons to service Kipushi. This equipment could either be sourced new from an overseas manufacturer (India or China) or be provided by establishing a PSP with Transnet to purchase and rehabilitate a portion of their existing 'B' fleet wagons.

The study has assumed a combination of containerised and break bulk concentrate out of either Durban or Richards Bay to China (Shanghai).

Figure 1.7 DRC to South Africa North–South Rail Corridor

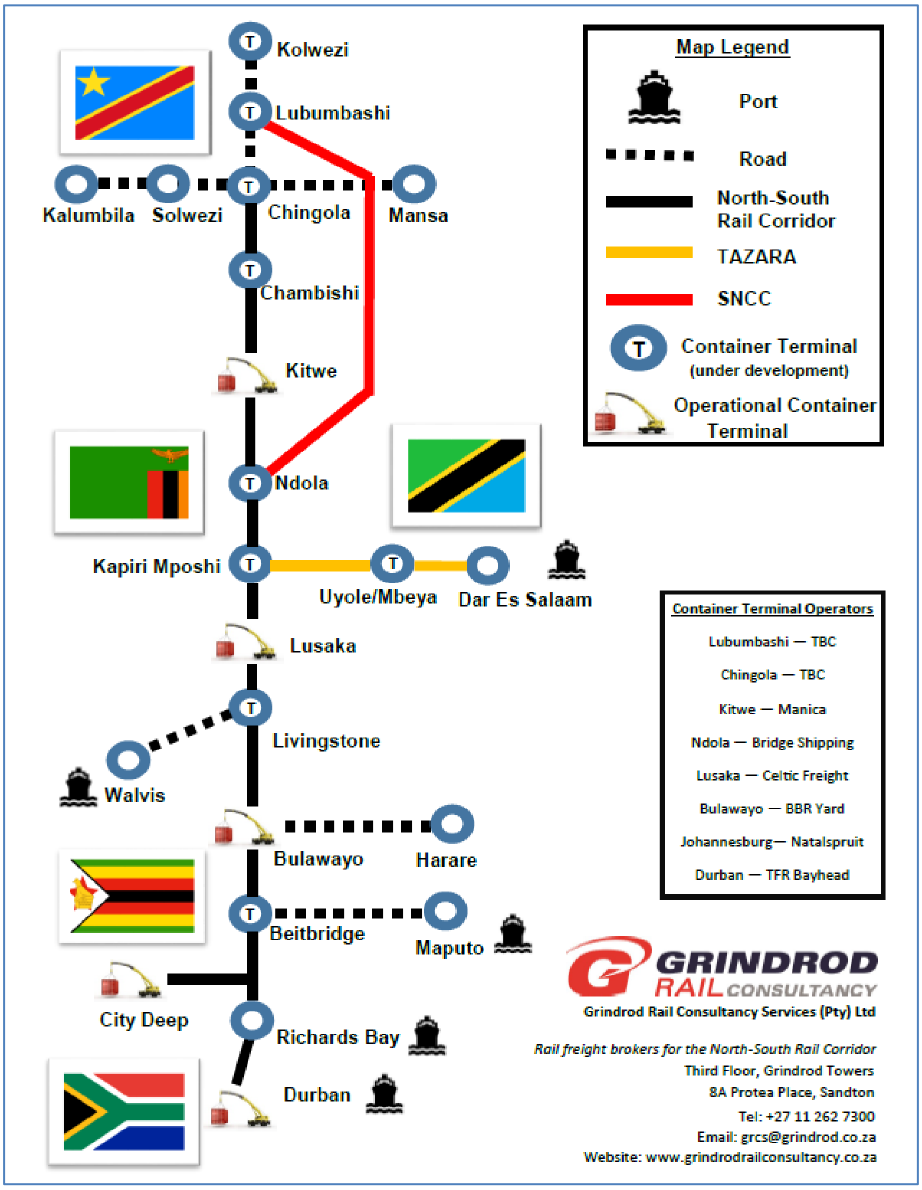


Figure by Grindrod, 2016.

1.11.8 Production

Future proposed mine production has been scheduled to optimise the mine output and meet the plant capacity. The mining production forecasts are shown in Table 1.13. Mine, process and concentrate production are shown in Figure 1.8 to Figure 1.10.

Table 1.13 Mining Production Statistics

Item	Unit	Total LOM	5 Year AVG	LOM Annual Average
Zinc Ore Processed				
Quantity Zinc Ore Treated	kt	8,581	777	780
Zinc Feed grade	%	32.14	30.20	32.14
Zinc Concentrate Recovery	%	89.61	88.76	89.61
Zinc Concentrate Produced	kt (dry)	4,196	354	381
Zinc Concentrate Grade	%	58.91	58.51	58.91
Metal Produced				
Zinc	kt	2,472	207	225

Figure 1.8 Mined Production

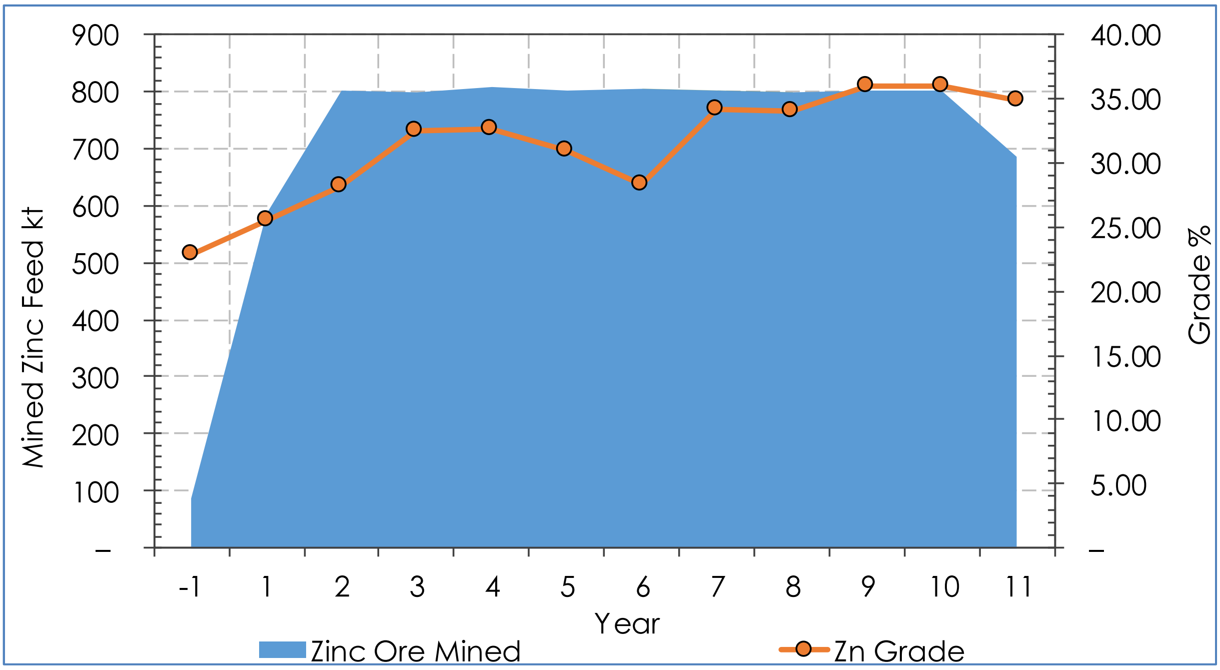


Figure by OreWin, 2017.

Figure 1.9 Process Production

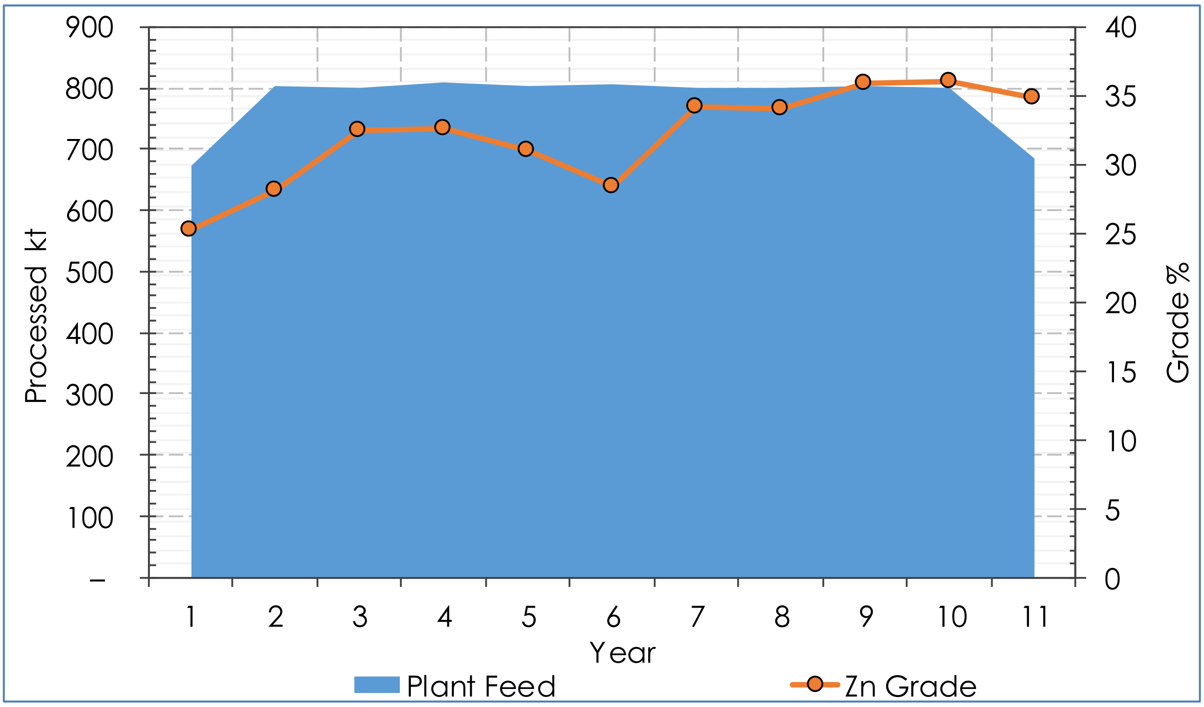


Figure 1.10 Concentrate and Metal Production

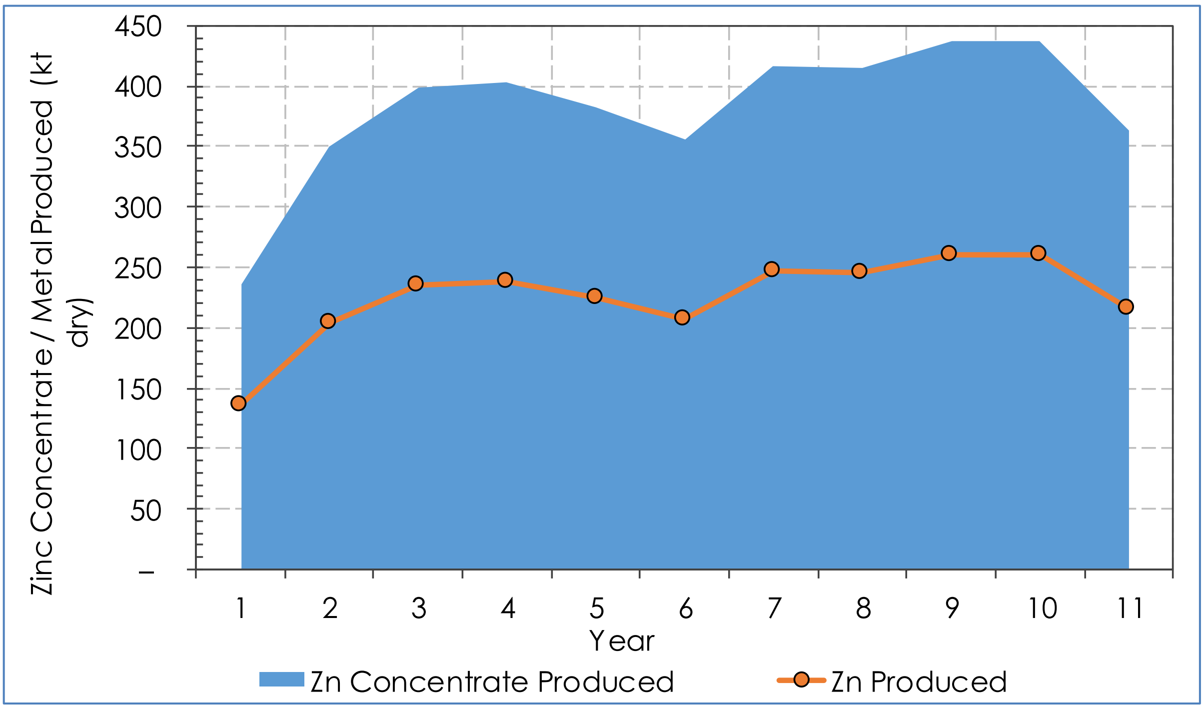


Figure by OreWin, 2017.

1.11.9 Economic Analysis

The estimates of cash flows have been prepared on a real basis as at 1 January 2018 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 1.14. The key economic assumptions for the discounted cash flow analyses are shown in Table 1.15. The results of $NPV_{8\%}$ sensitivity analysis to a range of zinc prices and discount rates is shown in Table 1.16. The results of $NPV_{8\%}$ and IRR sensitivity analysis to a range of zinc prices and zinc concentrate treatment charge is shown in Table 1.17.

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

Table 1.14 Financial Results

	Discount Rate	Before Taxation	After Taxation
Net Present Value (US$M)	Undiscounted	1,944	1,435
	5.0%	1,239	900
	8.0%	953	683
	10.0%	743	517
	12.0%	628	431
	15.0%	487	325
	18.0%	401	262
	20.0%	335	213
Internal Rate of Return	–	41.7%	35.3%
Project Payback Period (Years)	–	1.9	2.2

Table 1.15 Metal Prices and Terms

Parameter	Unit	Financial Analysis Assumption
Zinc Price	US$/lb	1.10
Zinc Treatment Charge	US$/t concentrate	170.00

Table 1.16 After Tax NPV_8 Sensitivity to Zinc Price and Discount Rates

Discount Rate (%)	Zinc (US$/lb)								
	0.80	0.90	1.00	1.10	1.20	1.40	1.50	1.70	2.00
Undiscounted	516	823	1,129	1,435	1,742	2,355	2,661	3,274	4,193
5%	254	472	687	900	1,111	1,533	1,744	2,165	2,796
8%	150	331	508	683	855	1,199	1,370	1,713	2,226
10%	96	257	414	568	719	1,021	1,172	1,473	1,923
12%	51	195	335	471	605	872	1,005	1,271	1,668
15%	-2	121	239	354	467	691	802	1,025	1,357
18%	-42	63	164	262	358	548	642	831	1,112
20%	-64	32	124	213	299	470	555	724	977

Note: Table shows NPV_8 $M.

Table 1.17 After Tax NPV_8 and IRR Sensitivity to Zinc Price and Zinc Treatment Charge

Zinc Treatment Charge (US$/t)	Zinc Price (US$/lb)								
	0.80	0.90	1.00	1.10	1.20	1.40	1.50	1.70	2.00
50.00	347	524	698	870	1,043	1,385	1,557	1,899	2,412
	23.1%	29.8%	35.8%	41.3%	46.5%	56.0%	60.5%	69.0%	80.5%
100.00	266	444	619	792	965	1,308	1,479	1,822	2,334
	19.8%	26.9%	33.2%	38.8%	44.2%	53.9%	58.4%	67.2%	78.8%
150.00	183	364	540	714	886	1,230	1,401	1,744	2,257
	16.3%	23.8%	30.4%	36.3%	41.8%	51.7%	56.4%	65.2%	77.1%
170.00	150	331	508	683	855	1,199	1,370	1,713	2,226
	14.9%	22.5%	29.2%	35.3%	40.8%	50.9%	55.5%	64.4%	76.4%
200.00	99	282	461	635	808	1,152	1,324	1,666	2,179
	12.6%	20.5%	27.4%	33.7%	39.3%	49.6%	54.3%	63.2%	75.4%
250.00	0	200	380	556	730	1,074	1,246	1,589	2,102
	8.0%	17.0%	24.4%	30.9%	36.8%	47.3%	52.1%	61.2%	73.6%

Note: Table shows NPV_8 $M and IRR.

Figure 1.11 Cumulative Cash Flow

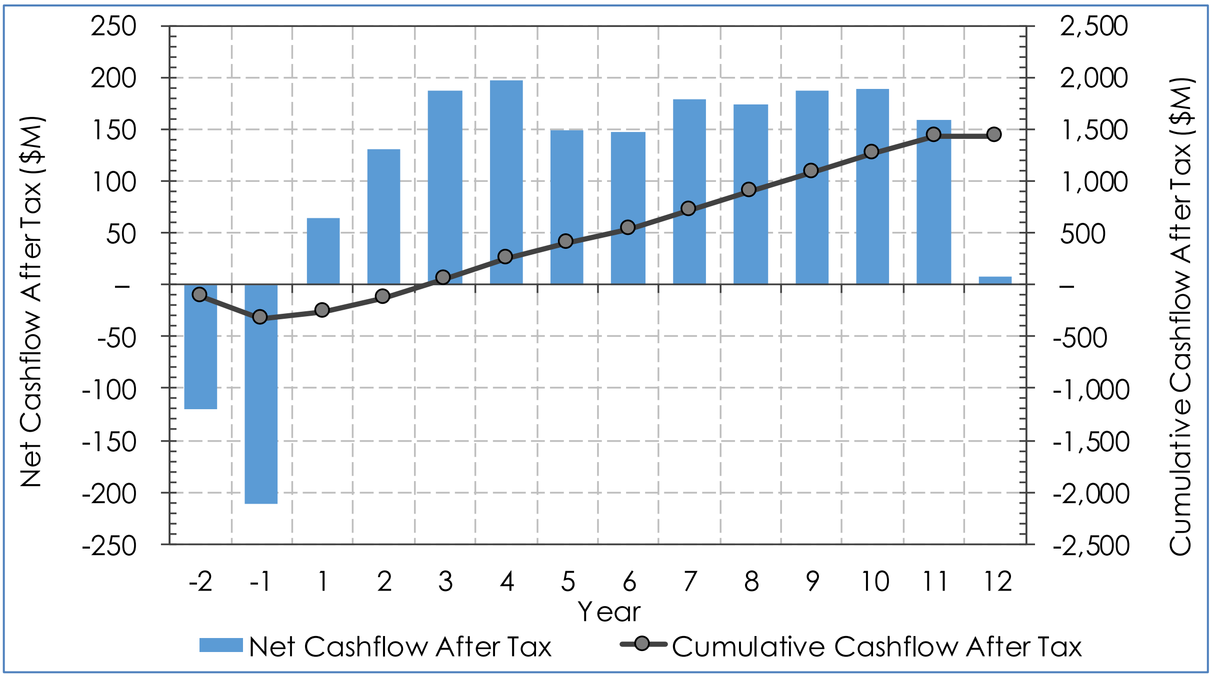


Figure by OreWin, 2017.

The total capital cost estimates for the Kipushi Project are shown in Table 1.18.

The estimated revenues and operating costs are presented in Table 1.19 along with the estimated net sales revenue value attributable to each key period of operation. The estimated cash costs are presented in Table 1.20.

Table 1.18 Total Project Capital Costs

Item	Pre-Production ($M)	Production ($M)	Total ($M)
Mining			
Underground Mine Refurbishment	17	–	17
Underground Mining	57	128	185
Capitalised Mining Operating Costs	37	–	37
Subtotal	**112**	**128**	**239**
Process and Infrastructure			
Process and Infrastructure	78	7	84
Rail	32	–	32
Capitalised Processing	7	–	7
Subtotal	**116**	**7**	**123**
Closure			
Closure	–	20	20
Subtotal	–	20	20
Indirects			
EPCM	12	–	12
Capitalised G&A	11	–	11
Subtotal	**23**	**–**	**23**
Others			
Owners Cost	11	–	11
Studies	5	–	5
Kico 2018 Site	33	–	33
Sustaining	–	24	24
Capital Cost Before Contingency	**300**	**178**	**478**
Contingency	37	–	37
Capital Cost After Contingency	**337**	**178**	**515**

Table 1.19 Operating Costs and Revenues

Description	Total ($M)	5-Year Average	LOM Average
		($/t Milled)	
Revenue			
Gross Sales Revenue	5,095	550	594
Less Realisation Costs			
Transport Costs	972	103	113
Treatment and Refining Charges	713	77	83
Royalties	197	21	23
Total Realisation Costs	1,883	202	219
Net Sales Revenue	3,212	348	374
Less Site Operating Costs			
Total Mining	415	52	48
Processing Zn	194	23	23
General and Administration	144	17	17
Total	753	93	88
Operating Margin ($M)	2,459	255	287
Operating Margin (%)	48.2	46.4	48.2

Table 1.20 Cash Costs

Description	5-Year Average	LOM Average
	US$/lb Payable Zn	
Mine Site Cash Cost	0.16	0.14
Realisation	0.34	0.35
Total Cash Costs Before Credits	0.50	0.48

1.12 Comparison to Other Projects

Using data for other zinc projects provided by Wood Mackenzie comparisons with the Kipushi 2017 PFS were made for the following results: contained zinc in Measured and Indicated Resource, production, capital intensity and C1 Cash Costs.

The Kipushi Project Mineral Resource Estimate, 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 1.12).

Figure 1.12 Top 20 Zinc Projects by Contained Zinc

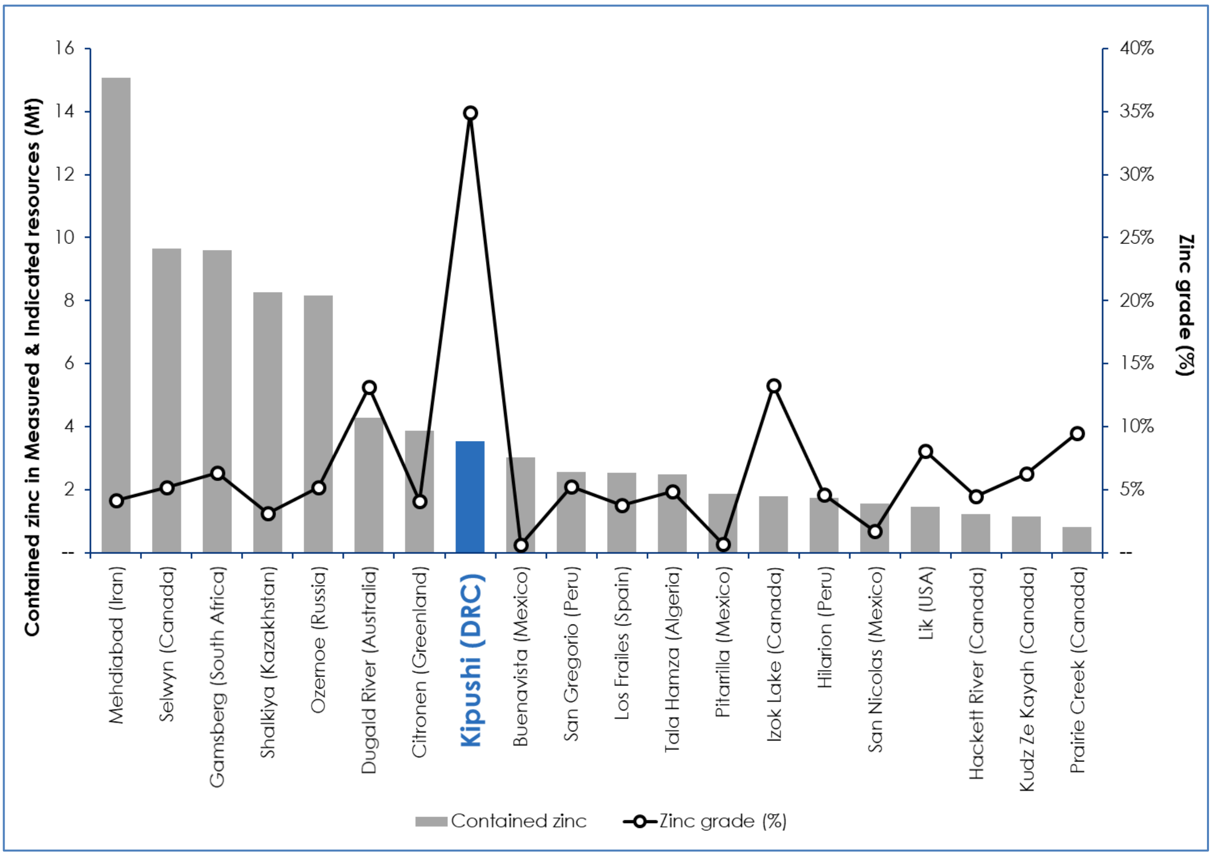


Figure by Wood Mackenzie, 2017.

Life-of-mine average planned zinc concentrate production of 381 ktpa, with a concentrate grade of 59% Zn, is expected to rank the Kipushi Project, once in production, among the world's major zinc mines (Figure 1.13). Based on research by Wood Mackenzie the world's major zinc mines defined as the world's 10 largest zinc mines ranked by forecasted production by 2018.

Figure 1.13 Major Zinc Mines Estimated 2018 Annual Zinc Production and Grade

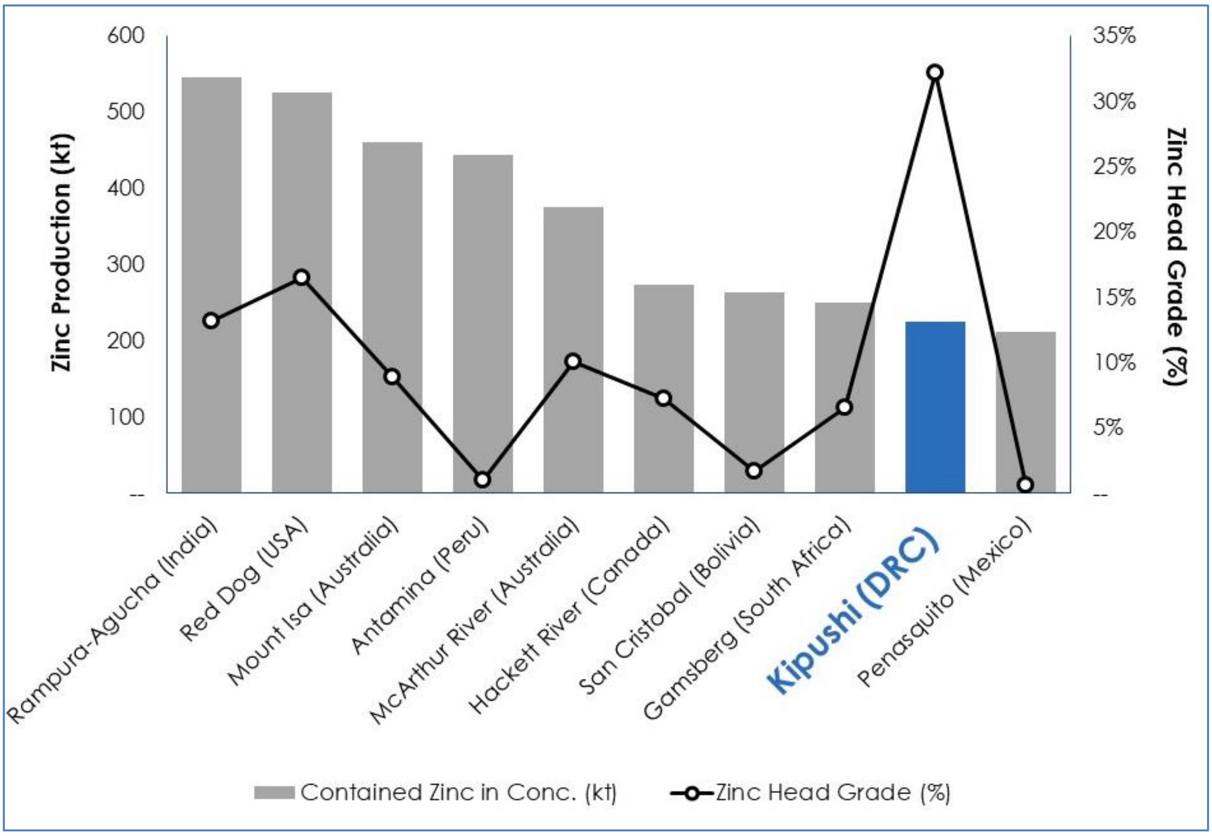


Figure by Wood Mackenzie, 2017.

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

Figure 1.14 Capital Intensity for Zinc Projects

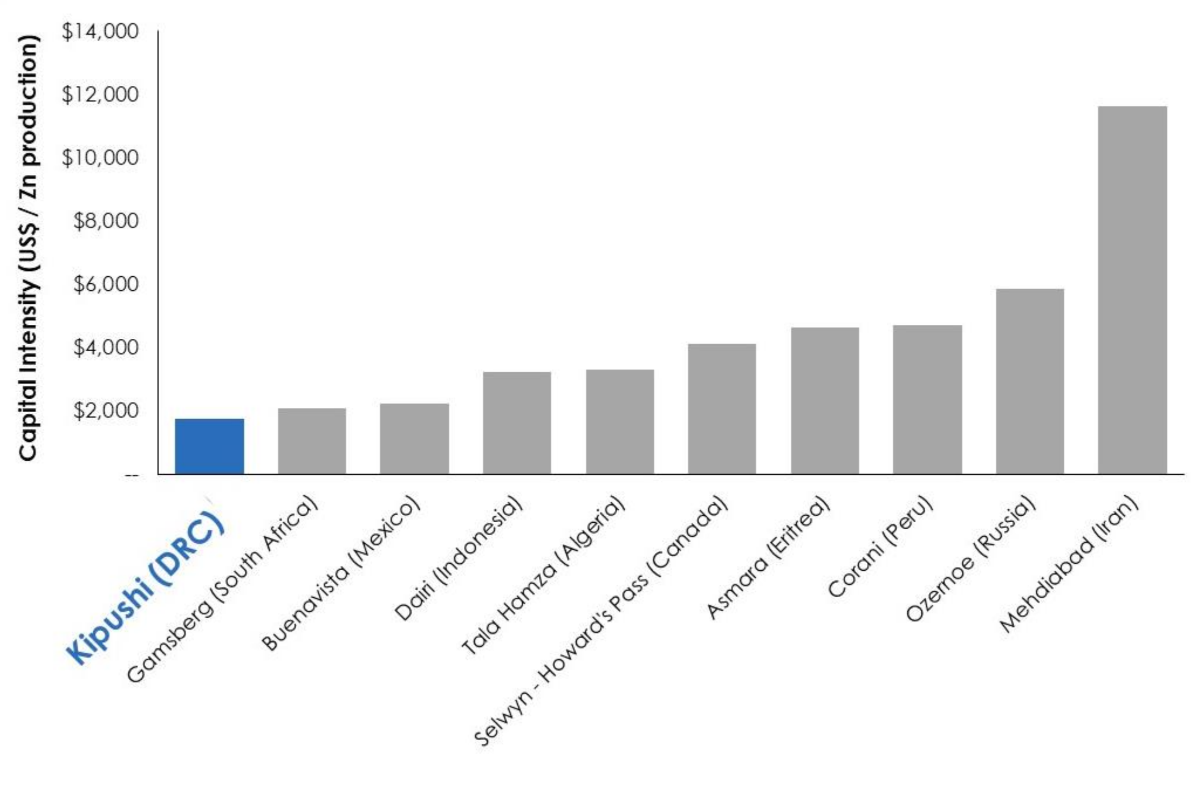


Figure by Wood Mackenzie, 2017.

Based on comparative data from Wood Mackenzie, C1 cash cost of US$0.54/lb of zinc is expected to rank the Kipushi Project, once in production, in the bottom quarter of the 2018 cash cost curve for zinc producers globally (Figure 1.15). Represents C1 cash costs which reflect the direct cash costs of producing paid metal incorporating mining, processing and offsite realisation costs having made appropriate allowance for the co-product revenue streams. Based on public disclosure and information gathered in the process of Wood Mackenzie's research.

Figure 1.15 2018 Expected C1 Cash Costs

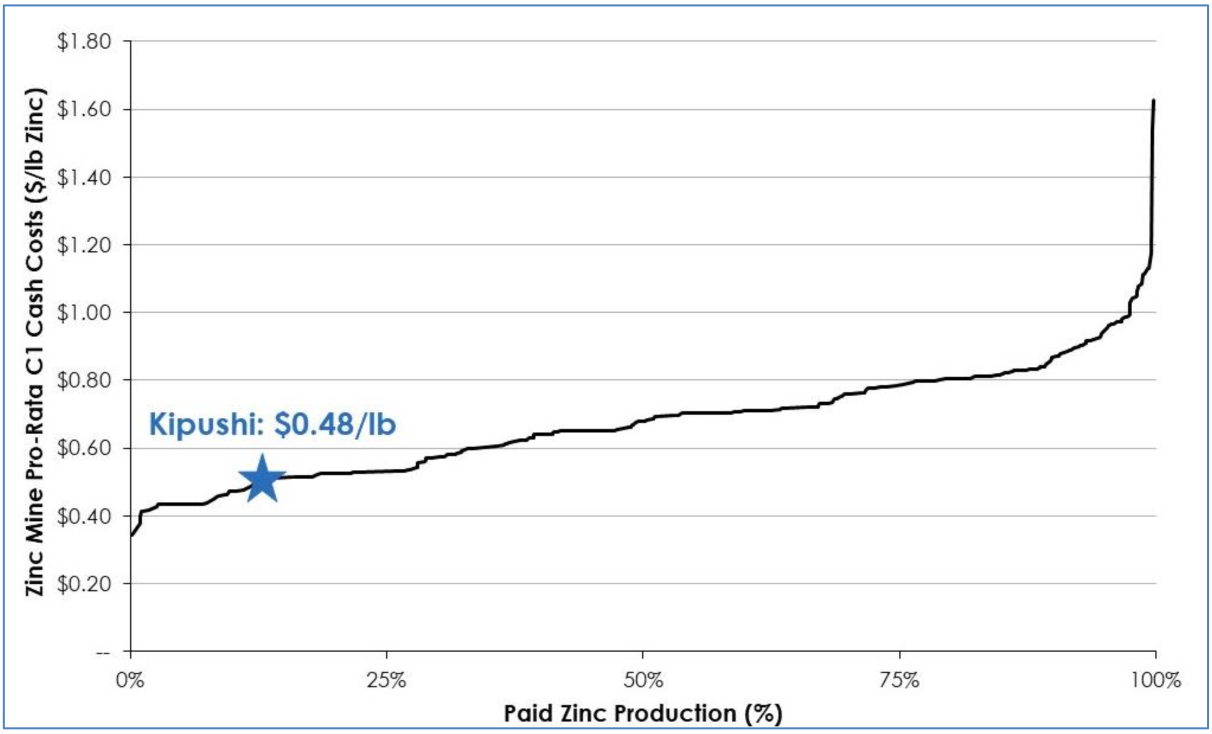


Figure by Wood Mackenzie, 2017.

1.13 Interpretation and Conclusions

The Kipushi 2017 PFS for the redevelopment of the Kipushi Mine is at a prefeasibility level of accuracy. It has identified a positive business case and it is recommended that the Kipushi Project is advanced to a 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 Project. The key areas for further work are:

1.13.1 Mineral Resources

- Further exploration work is not required in order to progress the project to a Feasibility Study. As part of the Feasibility Study, planning and costing of a grade control drilling programme, and other geological activities that will be required to support the mining operation, should be carried out.

1.13.2 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.

1.13.3 Mining

- Complete shaft and underground rehabilitation work.
- Additional study work to define the declines, ventilation, and material handling pass systems for FS.
- 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.

1.13.4 Process

- LOM grade and mineral variability needs to be defined at a more granular level to determine plant design/operating envelopes.
- Further metallurgical testwork including flowsheet optimisation.
- Variability testwork to review circuit performance for expected variations in feed concentrations.

1.13.5 Infrastructure

- Define the rail option development.
- Define what infrastructure should be demolished to make the mine safe and operable.
- Optimise surface infrastructure layout.
- Finalise location of the new tailings dam.

1.13.6 Marketing

- Investigate customer uptake for container transport.
- Investigate the optimal concentrate transport solution for bagging and bulk.

1.13.7 Environmental and Social

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

1.13.8 Project Financing

- Investigate financing options and sources.

Review of capital and operating cost estimates as part of the 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-Kakula Project; (ii) the Platreef Project, and (iii) the Kipushi 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 Project. Ivanhoe and Gécamines respectively own 68% and 32% of the Kipushi Project through Kipushi Corporation SA (KICO), the mining rights holder of the Kipushi Project.

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

2.2 Terms of Reference and Purpose of the Report

The Kipushi 2019 Resource Update includes restatement of the Kipushi 2017 Prefeasibility Study which includes the Kipushi Mineral Reserve from the Kipushi 2017 PFS. The Mineral Reserve in the Kipushi 2017 PFS remains valid. Further study work is currently incomplete and has not determined any results that require material changes to the Kipushi 2017 PFS.

The Kipushi 2019 Resource Update is an Independent Technical Report on the Kipushi Project prepared for Ivanhoe Mines Ltd. (Ivanhoe) as part of the strategy for redevelopment of the Kipushi Project.

The following companies have undertaken work in preparation of the Kipushi 2019 Resource Update and Kipushi 2017 PFS:

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

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

2.3 Qualified Persons

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

- Bernard Peters, B. Eng. (Mining), FAusIMM (201743), employed by OreWin as Technical Director - Mining was responsible for: Sections 1.1, 1.3, 1.4, 1.10, 1.11, 1.11.1, 1.11.2, 1.11.7, 1.11.8, 1.11.9, 1.12, 1.13.3. 1.13.6 to 1.13.8; Sections 2 to 5; Section 15; Sections 16, 16.2 to 16.16; Section 19; Section 20; Sections 21, 21.1 to 21.3, 21.5 to 21.6; Sections 22 to 24; Sections 25.3, 25.6 to 25.8; Section 26.3; Section 27.
- Michael Robertson, BSc Eng (Mining Geology), MSc (Structural Geology), Pr.Sci.Nat SACNASP, FGSSA, MSEG, MSAIMM, employed by The MSA Group (Pty) Ltd as a Principal Consulting Geologist was responsible for: Sections 1.5 to 1.8.2, 1.13, 1.13.1; Sections 2 to 3; Sections 6 to 12; Sections 25, 25.1; Section 26.2; Section 27.
- Jeremy Witley, BSc Hons (Mining Geology), MSc (Eng), Pr.Sci.Nat SACNASP, FGSSA, employed by The MSA Group (Pty) Ltd as a Principal Resource Consultant was responsible for: Sections 1.2, 1.13.1; Sections 2 to 3; Section 14; Section 25.1; Section 26.1; Section 27.
- William Joughin, FSAIMM (55634), employed by SRK Consulting (South Africa) (Pty) Ltd as Principal Consultant, was responsible for: Section 1.13.2; Sections 2; Sections 16.1; Section 25.2.
- Dean David, FAusIMM (CP) (102351), B App Sc (Metallurgy), employed by Wood (mining and Metals Australia West) as Technical Director – Process, was responsible for: Sections 1.9, 1.11.3 to 1.11.6, 1.13.4, 1.13.5; Section 13; Section 17; Section 18; Section 21.4; Sections 25.4, 25.5; Section 26.4; Section 27.

2.4 Site Visits and Scope of Personal Inspection

Site visits were performed as follows:

Mr Bernard Peters visited the Project from 1 June 2015 to 3 June 2015, 11 September 2015, on 24 October 2016 and from 26 to 28 June 2017. The site visits included briefings from geology and exploration personnel, site inspections of potential areas for mining, plant and infrastructure, discussions with other QPs and review of the existing infrastructure and facilities in the local area around the Project site.

Michael Robertson visited the Project from 20 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, from 11 May 2015 to 13 May 2015 and again from 13 November 2017 to 15 November 2017.

Mr William Joughin visited the project site from 19 May to 22 May 2014, from 27 November to 29 November 2017 and on 17 August 2018. The site visits included inspections of the drill core, underground visits to gain an impression of the ground conditions and discussions with the mine personnel on the local geology and previous mining activities conducted.

Dean David visited the Project from 5 October to 8 October 2018 and reviewed the site relative to current process and infrastructure concepts with special reference to impediments, limitations and opportunities. A basic level of familiarisation was gained of the geology of the deposit, especially the Big Zinc, and the processing characteristics of the ore as understood from testwork was confirmed through an underground visit and viewing significant drill core trays.

2.5 Effective Dates

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

- Effective date of the Report: 28 March 2019.
- Date of drillhole database close-out date for updated Mineral Resource estimate: 24 April 2018.
- Effective date of Mineral Resource update for mineralisation amenable to underground mining methods: 14 June 2018.
- Effective date of Mineral Reserves: 12 December 2017.

3 RELIANCE ON OTHER EXPERTS

The QPs, as authors of Kipushi 2019 Resource Update, have relied on, and believe there is a reasonable basis for this reliance, upon the following Other Expert reports as noted below. Individual QP responsibilities for the sections are listed on the Title Page.

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

3.1 Mineral Tenure

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

- KICO: report on the Kipushi Project Property Description and Location, March 2019.
- KICO: report on the Kipushi Project Property Description and Location, January 2018.
- A copy of the exploitation permit ("Certificat d'Exploitation") PE12434 dated 22 July 2011, issued by 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 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 Project through opinions and data supplied by KICO, and from information supplied by KICO staff. The QPs have fully relied on the following information provided by KICO in Section 4 and Section 20 Kipushi Environmental and Social Report, January 2018.
- Environmental Report on the Kipushi Zinc–Copper mine, Democratic Republic of Congo, by The Mineral Corporation, for Kipushi Resources International Limited (KRIL), 2007.
- Ivanhoe Mines Ltd., 2016: Kipushi Zinc Project – Preliminary Economic Assessment: unpublished letter prepared by representatives of Ivanhoe for OreWin, dated 12 May 2016.
- KICO: report on the Kipushi Project Property Description and Location, March 2019.
- KICO: report on the Kipushi Project Property Description and Location, January 2018.

3.3 Taxation and Royalties

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

- KICO: Email from KICO to OreWin on DRC Taxation for the Kipushi Project, November 2017.
- KPMG Services (Pty) Limited, 2016: Letter from M Saloojee, Z Ravat, and L Kiyombo to M Cloete, and M Bos regarding Updated commentary on specific tax consequences applicable to an operating mine in the Democratic Republic of Congo, dated 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.
- KICO: report on the Kipushi Project Property Description and Location, March 2019.

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

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

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

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

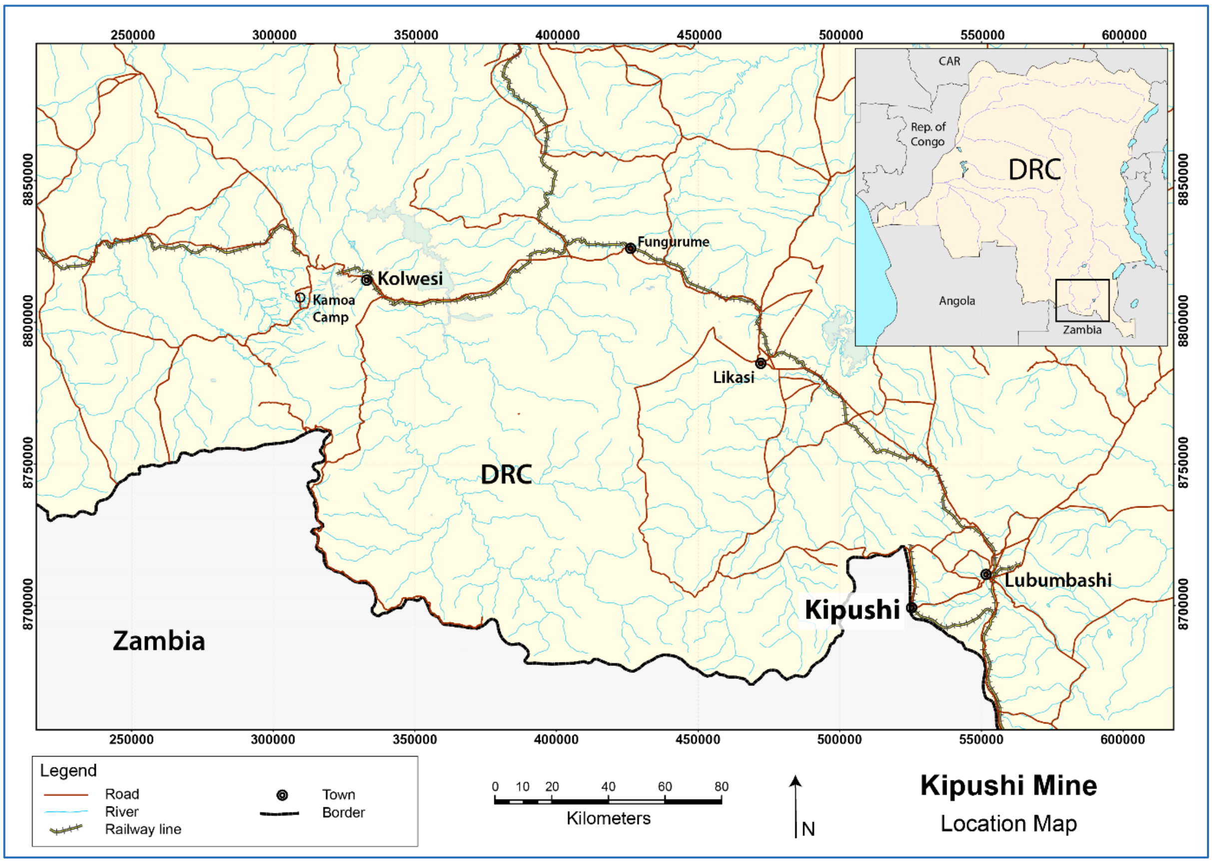


Figure by Ivanhoe, 2015.

4.2 Project Ownership

Kipushi Holding Ltd, a company registered under the laws of Barbados which is owned by Ivanhoe Mines Ltd. (Ivanhoe) (Kipushi Holding), and La Générale des Carrières et des Mines (Gécamines) have a joint-venture agreement (JV Agreement) over the Kipushi Project. Kipushi Holding and Gécamines respectively own 68% and 32% of the Kipushi Project through Kipushi Corporation (KICO) which holds the mining right required for the implementation of this project.

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

4.3 Mineral Tenure

KICO holds the exclusive right to engage in mining activities within the Kipushi Project area through a mining right, Exploitation Permit No. 12434 (PE12434), valid until 3 April 2024 and covering 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.

The Exploitation Permit No. 12434 resulted from the partial transfer of Exploitation Permit No. 481 previously held by Gécamines, was granted by Ministerial Order No. 0290/CAB.MIN/MINES/01/2011 dated 02 July 2011 and is evidenced by Exploitation Certificate No. CAMI/CE/6368/11 dated 22 July 2011, and granted KICO the exclusive right to perform exploration, development and exploitation works concerning silver, cobalt, cooper, germanium, and zinc.

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

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

The boundaries of Exploitation Permit No. 12434, indicated in the Exploitation Certificate, cross the international border, as do some of the 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.

Table 4.1 Boundary Coordinates for Permit Comprising the Kipushi 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.

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

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

4.4 Surface Rights

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

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

- Enter into the exploitation perimeter to proceed to mining operations.
- Build the facilities and infrastructure necessary for mining exploitation.
- Use water and wood resources located within the mining perimeter, for the needs of the mining exploitation, subject to compliance with the norms defined in the relevant Environmental Impact Study (EIS) and Project Environmental Management Plan (PEMP).
- Proceed to the works of extension of the mine.

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

However, there are a number of exceptions, agreed between KICO and Gécamines and to be interpreted restrictively. Gécamines remain the owner, on the basis of specific land rights to be established in favour of Gécamines, of:

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

In addition, a number of assets defined as being the Rented Facilities and Equipment (Installations et Equipements Loués), described in Appendix C of Amendment No. 3 to the JV Agreement, are rented by Gécamines to KICO under a lease agreement that was the subject of a settlement agreement dated 14 June 2013.

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

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

Discussions with Gecamines concerning the surface facilities required for the development of the Kipushi Project are planned in 2018.

The current Kipushi Mine layout is shown in Figure 4.2.

Figure 4.2 Kipushi Existing Mine Layout

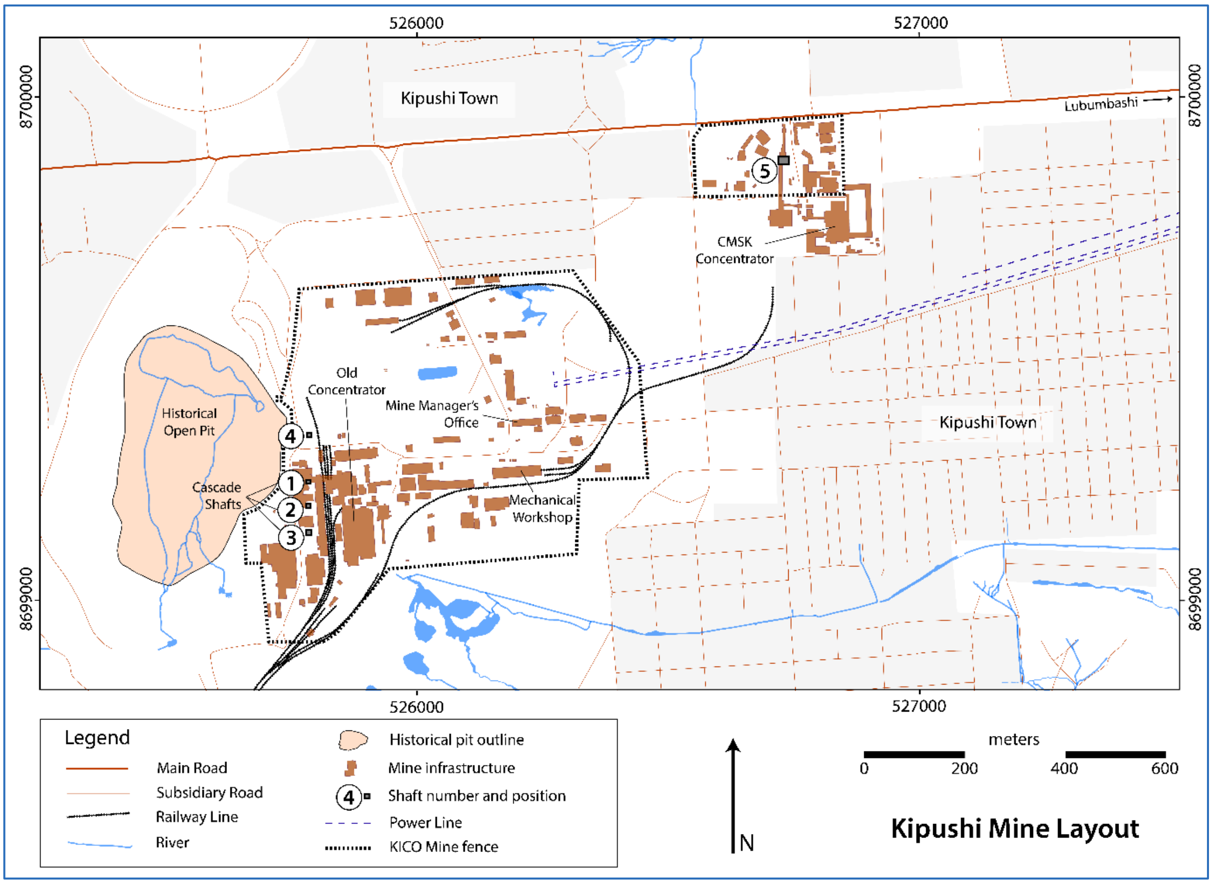


Figure by Ivanhoe, 2015.

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 Mining Code. As Exploitation Permit No. 12434 is under Force Majeure, KICO will pay these fees only when the Force Majeure will be lifted.

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

All payments relating to Exploitation Permit No. 12434 and agreements associated with the Kipushi Project have been made and Exploitation Permit No. 12434 is held in good standing.

4.6 Environmental Liabilities

The property covered notably by Exploitation Permit No. 12434 was the subject of an environmental audit by the Department in Charge of the Protection of the Mining Environment (DPEM) within the Ministry of Mines in August 2011. DPEM subsequently granted Gécamines a release of its environmental obligations over the perimeter covered notably by Exploitation Permit No. 12434. KICO commissioned a summary environmental liabilities assessment study which was completed in August 2012 by Golder Associates. It serves as an environmental snapshot as to the state of the property when Kipushi Holding acquired the Kipushi Project in November 2011.

KICO is currently in the process of revising the Project EIS and PEMP.

4.7 Mining Legislation in the DRC

4.7.1 Mineral Property and Title

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

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

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

The Minister of Mines supervises, without limitation, the Cadastre Minier (DRC mining registry), the Departments of Mines and Geology and the Department in charge of the protection of the mining environment.

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

- The Prime Minister, who is notably responsible for enacting the Mining Regulations for the implementation of the Mining Code and declaring mineral substances as being a strategic mineral substance.
- The Prime Minister exercises his rights by decrees adopted in Council of Ministers, upon proposal of the Minister of Mines and, where appropriate, the relevant Ministers.
- The Minister of Mines, who has notably jurisdiction over the granting, refusal and withdrawal of mining rights.
- The *Cadastre Minier*, is a public entity supervised by the Minister of Mines that is notably responsible for the management of the mining domain and mining rights. It conducts, without limitation, administrative proceedings concerning the application for, and registration of, mining rights, as well as the withdrawal and expiry of those rights.
- The Department of Minesis notably responsible for controlling and monitoring the performance of activities regarding relation to mines in accordance with legal and regulatory provisions in force.
- The Department in charge of the protection of the mining environment is notably responsible, in collaboration with the Congolese Agency for Environment, the regulation national fund of promotion and social service and, where appropriate, any other relevant body of the State, for implementing the mining regulations concerning environment protection and performing the environmental examination of environmental and social impact studies, and environmental and social management plans. These administrations are also notably responsible for controlling and monitoring, without limitation, the obligations of the holders of mining rights concerning health and safety and the protection of environment in the sector of mines; and

- The Chief of the Provincial Department of Mines also has, without limitation, authority to control and monitor mining activities in Province.

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

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

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

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

The DRC is divided into mining cadastral grids using a WGS84 Geographic coordinate system outlined in the Mining Regulations. This grid defines uniform quadrangles, or cadastral squares, typically 84.95 ha in area, which can be selected as a "Perimeter" to a mining right. A perimeter under the Mining Code is in the form of a polygon composed of entire contiguous quadrangles subject to the limits relating to the borders of the National Territory and those relating to prohibited and protected reserves areas as set forth in the Mining Regulations.

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

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

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

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

Law No.18/001 dated 09 March 2018 amending and completing the 2002 Mining Code brought significant changes to the legal regime governing mining activities, including, without limitation, numerous issues, such as:

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

  Since the the enactment of Law No.18/001, the more stringent tax requirements in Law No.18/001 apply to all mining companies, including KICO,

  With regard to the current contrary interpretation of DRC, in spite of the requests made by KICO to have the stability guarantee respected by DRC and all its administrations during the stabilised period, KICO proceeds to the payment of the taxes required by DRC administrations, under duress and for the sole purpose of preventing, as far as possible, the damages that could result from sanctions imposed on KICO. KICO already received tax adjustments for lack of compliance with new requirements on environment and expatriate taxes that are, in its view, not applicable to KICO. KICO challenged such tax adjustments and will continue to challenge such tax adjustments to preserve its rights;

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

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

  Significant taxes that should not be applicable to KICO with regard to the stability guarantee it is entitled to are nevertheless applied by DRC administrations to KICO, for instance in relation to environmental taxes, expatriate taxes or explosives. They will increase the Kipushi project's costs.

Also see the comments below concerning royalties;

- Increased importance of the commitments made vis-à-vis local communities on social and environmental aspects, the respect of the commitments made concerning social obligations in accordance with the schedule set out in the cahier des charges to be negotiated and entered into being a new condition to maintain the validity of the mining rights. Law No.18/001 also inserted an obligation to pay an annual contribution of 0.3% of the turnover for community development projects;

- Increased requirements concerning local procurement insofar as pursuant to the Mining Code, subcontractors, in the meaning of the Mining Code, must be DRC legal entities with Congolese financing ("à capitaux congolais"). Subject to further clarifications to be adopted, KICO understands from the recitals of Law No.18/001 that it means DRC companies having the majority of their share capital being directly held by Congolese individuals.

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

Pursuant to the 2017 Subcontracting law, subcontracting, in the meaning of the 2017 Subcontracting Law (which is distinct from the definition resulting from the Mining Code is an activity reserved to businesses with Congolese, financing, promoted by Congolese and having their head office in DRC. However, when there is non-availability or non-accessibility of the above expertise and subject to providing evidence to the relevant authority, the main contractor is authorised to enter into an agreement with any other Congolese or foreign business for a maximum duration of six months. The sectorial Minister or local authority must be informed previously. Subcontracting, in the meaning of the 2017 Subcontracting Law, is limited to a maximum of 40% of the global value of a contract. In addition, the main contractor is not authorised to oblige the subcontractor, in the meaning of Subcontracting Law, to totally prefinance the cost of the subcontracted operation or activity and must pay, before the beginning of the works, an advance payment covering at least 30% of the subcontracting contract. Any subcontracting above a threshold of approximately $60,400 requires a public tendering process (*appel d'offres*). Fines for non-compliance with the 2017 Subcontracting Law are significant. KICO is therefore in the process of ensuring that all its subcontractors, in the meaning of the Mining Code, comply with the requirements of the 207 Subcontracting Law.

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

KICO is nevertheless doing its best efforts to voluntary ensure compliance with the new requirements, as well as ongoing improvement in this respect to favour the development of local subcontractors, in the meaning of the Mining Code, as well as the selection of local subcontractors, in the meaning of the 2017 Subcontracting Law. Thus, KICO already adopted voluntarily several measures since the entry into force of the new legal framework governing mining activities in March 2018 and is currently in the process of finalising the development of its related action plan to mitigate, as far as possible, associated risks. KICO will also monitor the regulatory provisions to be adopted to ensure, as far as possible, adequate enforcement of the relevant legislative requirements.

There are also in the 2017 Subcontracting Law requirements applicable to all companies, for instance, an obligation to publish each year the list of the subcontractors, in the meaning of the 2017 Subcontracting Law, and to implement, within the companies, a training policy enabling Congolese to acquire the technicity and qualification required for the performance of some activities. KICO is in the process of performing those obligations in spite of the numerous uncertainties resulting from a lack clarity of the implementing regulations;

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

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

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

Pursuant to Decree No.18/042 dated 24 November 2018, cobalt, germanium and colombo-tantalite "coltan" were declared as being "strategic mineral subtances"; and

- The Mining products for sale that must be compliant with the nomenclature set out by the applicable regulations.

Pursuant to Article 7 of the interministerial order No. 0129/CAB.MIN/MINES/01/2017 and 032/CAB.MIN/FINANCES/2017 regulating the trading and export of mining products for sale, the export of copper concentrates is prohibited. However, a moratorium was granted until the definitive resolution of the energy deficit, to all mining operators who produce copper concentrate. The grades of such concentrate must comply with the values indicated in the table appended to this interministerial order.

This nomenclature could be changed anytime by the Ministry of Mines, in collaboration with the Ministry responsible for Foreign Trade.

There are also a number of new requirements, such as the obligation to build a building for the registered office, the obligation to have a share capital reaching at least 40% of the required financial resources or distinct mines that remain unclear. KICO is still in the process of assessing whether or not they should apply to KICO. Subject to further analysis and verification and to contrary interpretation from the DRC government authorities, KICO's preliminary view is that those new requirements should not apply to KICO.

KICO, considers that KICO should be protected against most adverse changes impacting the rights attached to its mining rights, including the right to export mining products and the tax regime applicable to such mining rights with regard to the 10-year stability guarantee KICO is entitled to in accordance with Article 276 of the Mining Code and the share transfer agreement entered into between DRC and Kamoa Holding. They nevertheless note the current contrary interpretation adopted by DRC administrations.

The economic analysis of this Kipushi 2019 Resource Update has not been changed from the Kipushi 2017 Prefeasibility Study and remains the most current study work available. Further study work is currently incomplete and has not been updated with the revised royalty and tax rates.

4.7.3 Exploitation Permits

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

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

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

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

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

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

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

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

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

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

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

4.7.4 Sale of Mining Products

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

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

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

The authorisation of the DRC Minister of Mines is required under the Mining Code for exporting unprocessed ores (minerais à l'état brut) for processing outside the DRC. This authorisation will only be granted if the holder who is applying for it demonstrates at the same time:

- The inexistence of a possibility to process 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.

4.7.5 Surface Rights Title

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

The soil is the exclusive, non-transferable and lasting ownership of the DRC State (Law No. 73-021 dated 20 July 1973, as amended by Law No. 80-008 dated 18 July 1980). However, the DRC State can grant surface rights to private or public parties. Surface rights are distinguished from mining rights, since surface rights do not entail the right to exploit minerals or precious stones. Conversely, a mining right does not entail any surface occupation right over the surface, other than that required for the operation.

The Mining Code provides that subject to the potential rights of third parties over the relevant soil, the holder of an exploitation mining right has, with the authorisation of the Governor of the relevant Province, after opinion from the relevant department of the Administration of Mines notably within the perimeter of the mining right, the right to occupy the parcels of land required for its activities and the associated industries, including the construction of industrial facilities, dwellings and facilities with a social purpose, to use underground water, the water from non-navigable, non-floatable watercourses, notably to establish, in the context of the concession of a waterfall, an hydroelectric power plant aimed at satisfying the energy needs of the mine, to dig canals and channels, and establish means of communication and transport of any type. KICO was granted with such an authorisation from the Governor of the Province on 23 July 2014.

KICO nevertheless noted a typo in one of the mining rights referred to in the above mentioned authorisation and is in the process of preparing an interpretative letter to ensure as soon as possible that the Province Governor's authorisation adequately covers the perimeter of Exploitation Permit No. 13025.

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

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

4.7.6 Royalties

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

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

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

Amendments to the 2002 Mining Code were nevertheless adopted by the above mentioned Law No.18/001 dated 09 March 2018.

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

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

At the date of this Report, the zinc concentrate that KICO intends to sale and export is not listed among the strategic mineral substances.

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

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

4.7.7 VAT Exoneration

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

4.7.8 Environmental Obligations

All mining operations must have an approved environmental plan and the holders of the right to conduct such operations are responsible for compliance with the rehabilitation requirements provided in the plan. When applying for an exploitation permit, a company must complete an environmental impact study (EIS) to be filed with the Project Environmental Management Plan (PEMP).

The Mining Code provides for additional environmental requirements, including the obligation to file a financial guarantee for rehabilitation, etc. Funds posted as financial guarantee are not at the disposal of the Department in charge of the protection of the mining environment of the Ministry of Mines and are to be used for the rehabilitation of a mining site.

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

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

The Mining Code also requires an environmental audit every two-year period as from the date of approval of the initial project EIS.

Breaches with environmental obligations can lead to significant sanctions, including suspension of mining activities and confiscation of the financial security.

Upon mine closure, shafts must be filled, covered or enclosed. After a closure environmental audit and an in-situ audit by the DPEM, a certificate of release of environmental obligations can be obtained.

In accordance with the above-mentioned obligation, KICO is in the process of revising in the Project EIA and environmental management plan and is also performing an ongoing compliance audit to ensure full compliance with its environmental obligations.

4.7.9 Surface Rights

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

In order to access the surface infrastructure, KICO has entered into a lease agreement with 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 shall pay rental fees of $30,000 per month to lease the areas required for its operations.

The payment of those rental fees to Gécamines is currently blocked in accordance with a court decision relating to a dispute between Gécamines and a Gécamines' creditor. However, the relevant amounts must be blocked by KICO so that KICO can pay the relevant entity to be determined by DRC Courts.

Discussions with Gécamines are on going in relation to surface facilities.

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

The Katanga region occupies a high plateau covered largely by Miombo (Brachystegia sp.) woodland and savannah. Kipushi lies at approximately 1,350 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 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.

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

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

KICO has a significant amount of underground infrastructure at the Kipushi Project, including a series of vertical mine shafts to various depths, associated head frames, and accompanying underground mine excavations. The newest shaft (Shaft 5) is 8 m in diameter and 1,240 m deep, with a lowest operating level of 1,150 mRL. It provides the primary access to the lower levels of the mine, including the Big Zinc. It has three independent friction hoists, and all compartments remain operational. The condition of the facility is fair but will require a refurbishment 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 which has been exposed since dewatering, is in relatively good order. The crusher is being replaced as the cost of refurbishment was determined to exceed the replacement cost.

The Kipushi 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. Gécamines owns all of the surface infrastructure.

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 the 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 Project would require refurbishment of underground access via Shaft 5, and construction of new ore processing and waste disposal facilities. Process water for any planned mining operation could be sourced from the underground pumping operations.

5.4 Surface Rights

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

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

6 HISTORY

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

A Belgian company, Union Minière du Haut Katanga (UMHK) started prospecting in the area in 1922 and commenced production in 1924. UMHK reportedly operated on a more or less uninterrupted basis for 42 years, initially by open pit until 1926 and subsequently by the underground methods of sub-level caving and sub-level stoping. The mine was originally known as the Prince Leopold Mine. In 1966, with the formation of the State-owned mining company 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. The mine was then put on care-and-maintenance due to a lack of hard currency to purchase supplies and spares.

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

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

The Big Zinc, adjacent to the Fault Zone on the footwall side, was discovered shortly before the mine suspended production, and had 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 holds the remaining 32% interest in Kipushi.

Prior to closure, the Kipushi deposit had 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.

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

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

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

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

The previous Technical Report was the Kipushi 2017 Prefeasibility Study which presented the results of exploration drilling, mineral resource estimation, and mine planning on the Big Zinc for the redevelopment of the Kipushi Project.

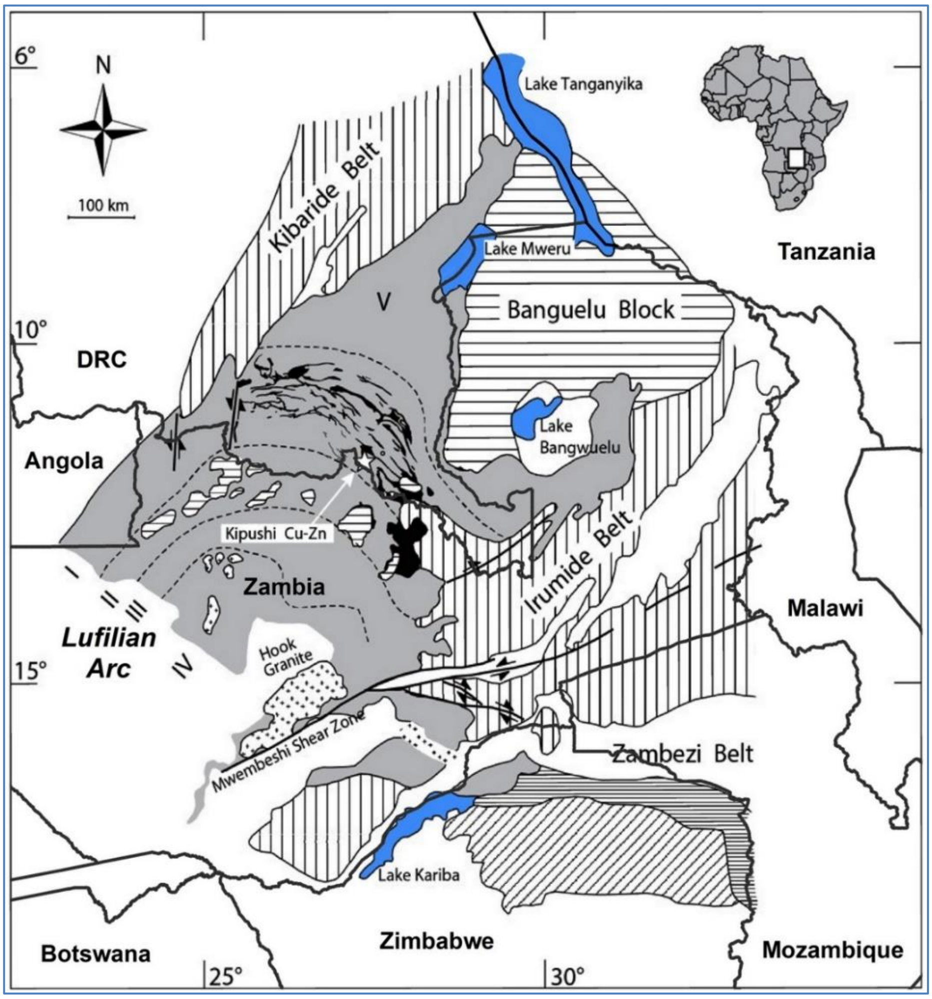
7 GEOLOGICAL SETTING AND MINERALISATION

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

7.1 Regional Geology

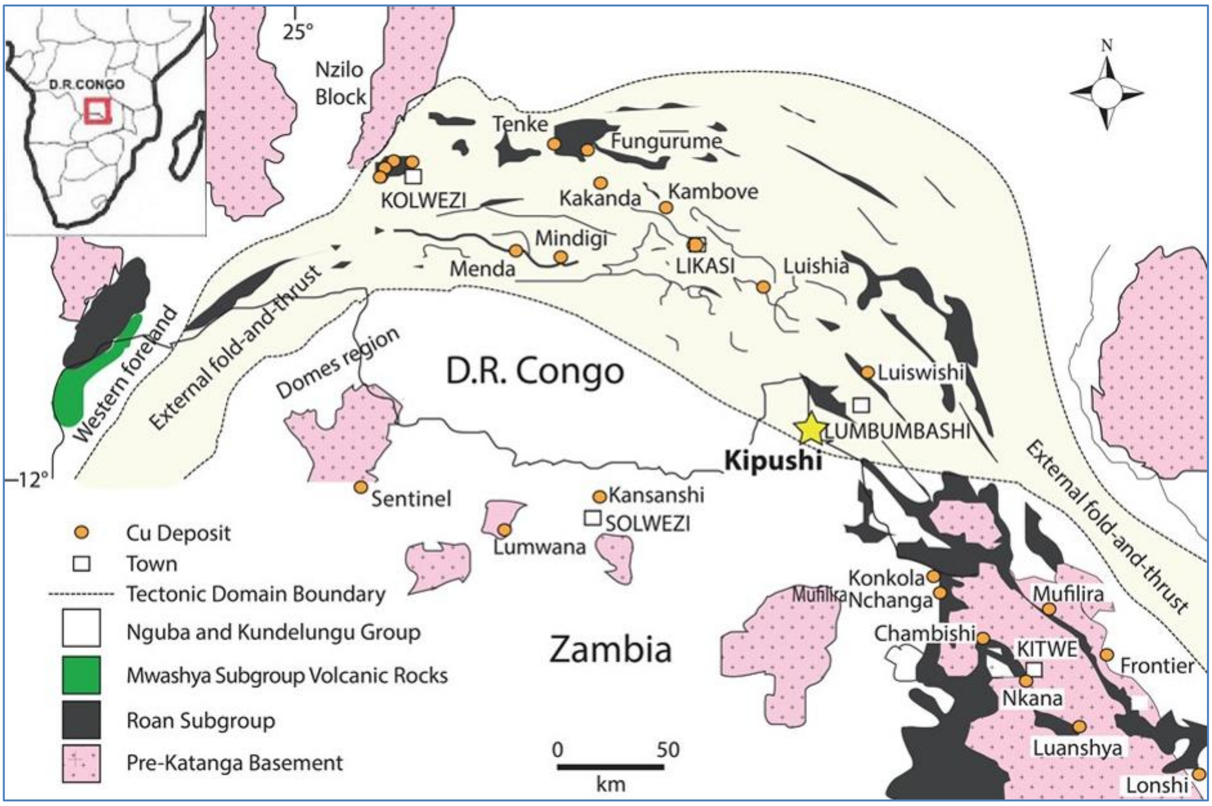
Kipushi is located within the Central African Copperbelt a northerly convex arc extending approximately 500 km from north central Zambia through the southern part of the DRC into Angola (Figure 7.1). The Central African Copperbelt constitutes a metallogenic province that hosts numerous world-class copper-cobalt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

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



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 Project



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

The Central African Copperbelt is contained in the Katangan basin, an intracratonic rift that records onset of growth at ~840 Ma and inversion at ~535 Ma (Selley et al., 2018). The lowermost sequences were deposited in a series of restricted rift basins that were then overlain by laterally extensive, organic rich, marine siltstones and shales. This horizon is overlain by what became an extensive sequence of mixed carbonate and clastic rocks of the Upper Roan Group (Selley et al., 2005).

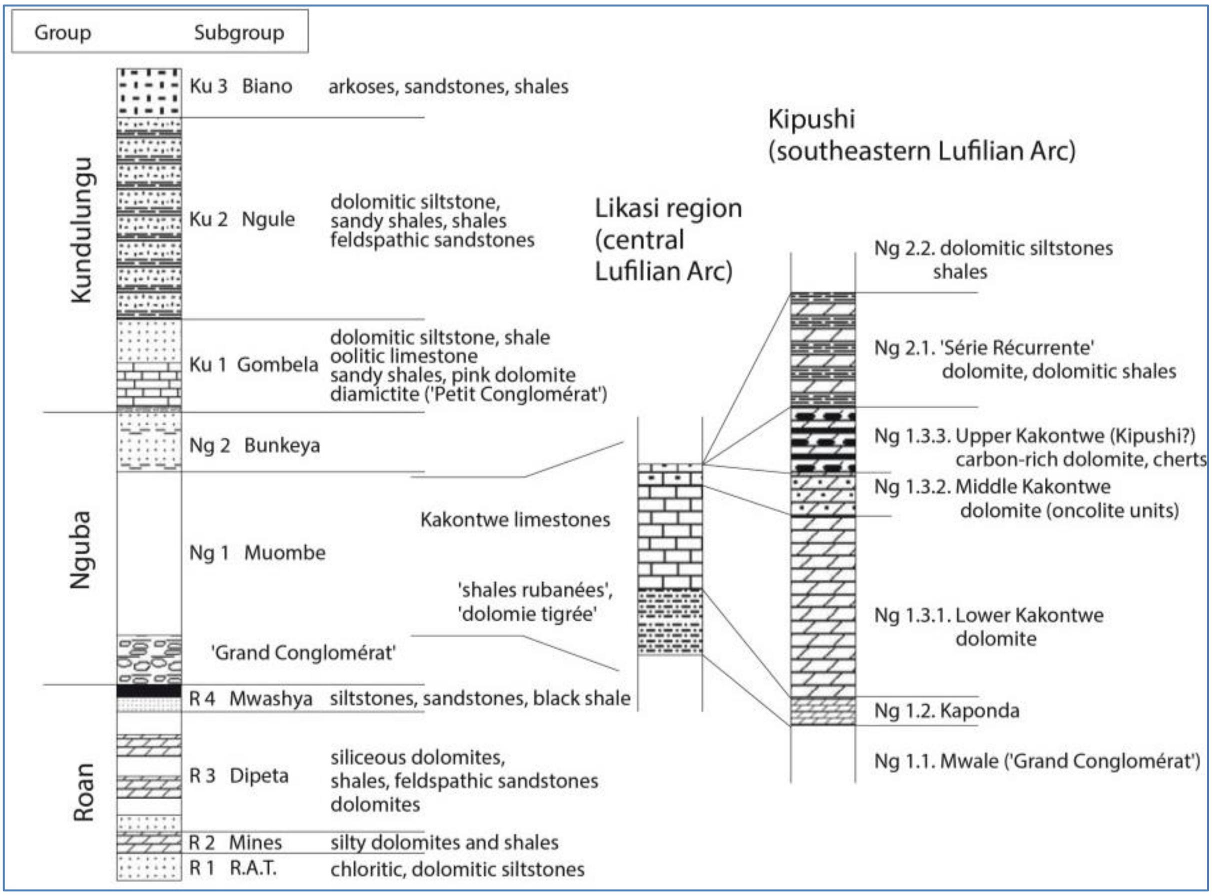
The extensional geometry was preserved through orogenesis, forming what is known as the Lufilian Arc. The arc geometry, similar in character to oroclinal bending, has conventionally been interpreted to be composed of a stack of thin-skinned, north-verging fold and thrust sheets (e.g., François and Cailteux, 1981; Kampunzu and Cailteux, 1999), however other work (De Magnee and François,1988; Jackson et al., 2003; Selley et al., 2018) favours a salt tectonic origin for the Copperbelt geometry. The crustal scale Mwembeshi Dislocation Zone separates the Lufilian Arc from the Zambezi Belt to the south.

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

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

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

7.1.2 Mineralisation and Tectonic Evolution

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

Mineralising fluids appear linked to residual evaporitic brines generated during deposition of the basin-wide salt-sheets, occupying large sub and intrasalt aquifers from ~800 Ma. This marks the earliest likely mineralising event, particularly in the Zambian-type stratiform

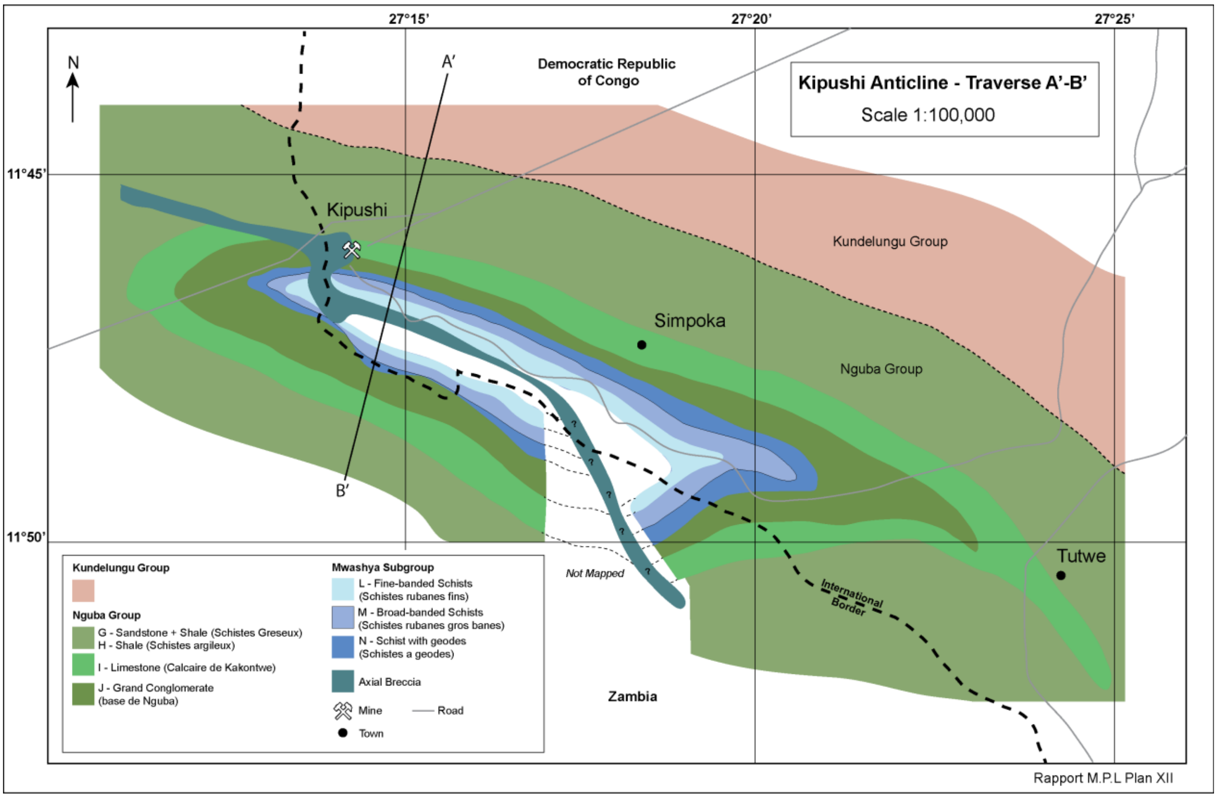
Cu ± Co ores (Selley et al., 2018). At variable times from ~765 to 740 Ma, movement in the salt sheets in the Congolese part of the basin caused their modification allowing deeper-level residual brines to interact with reducing elements and form the stratiform ores (Selley et al., 2018).

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

7.1.3 Structure

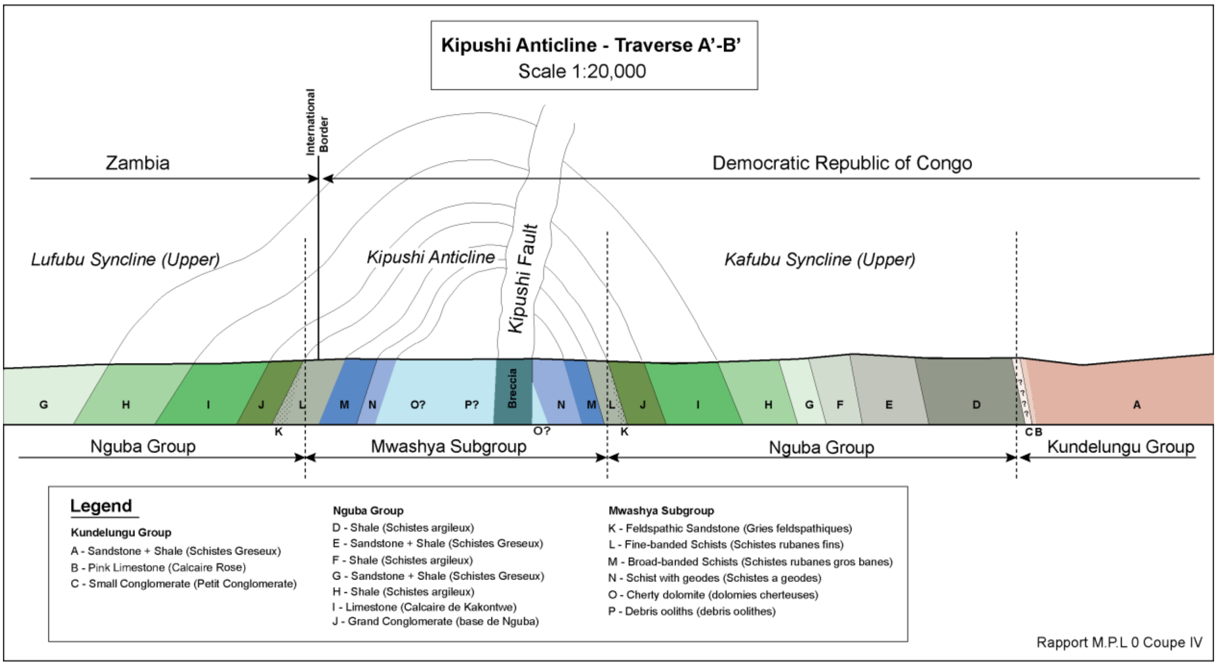
The Kipushi Project is located on the northern limb of the regional west-north-west trending Kipushi Anticline, which straddles the border between Zambia and the DRC. The northern limb of the anticline dips at 75–85° to the north–north-east, and the southern limb at 60–70° to the south–south-west, as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

Figure 7.4 Geological Map of the Kipushi Anticline



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

Figure 7.5 Section Through the Kipushi Anticline



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

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

Figure 7.6 Interbedded Dolomite-shale/Siltstone Unit in the Upper Kakontwe Formation at 153 m in KPU070 (hole orientation -35 to 125). Bedding Dips Steeply to North-North-West (here in proximity to Kipushi Fault Zone) 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 Local Geology

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

The only surviving production-era geological maps at 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.

7.2.1 Stratigraphy

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

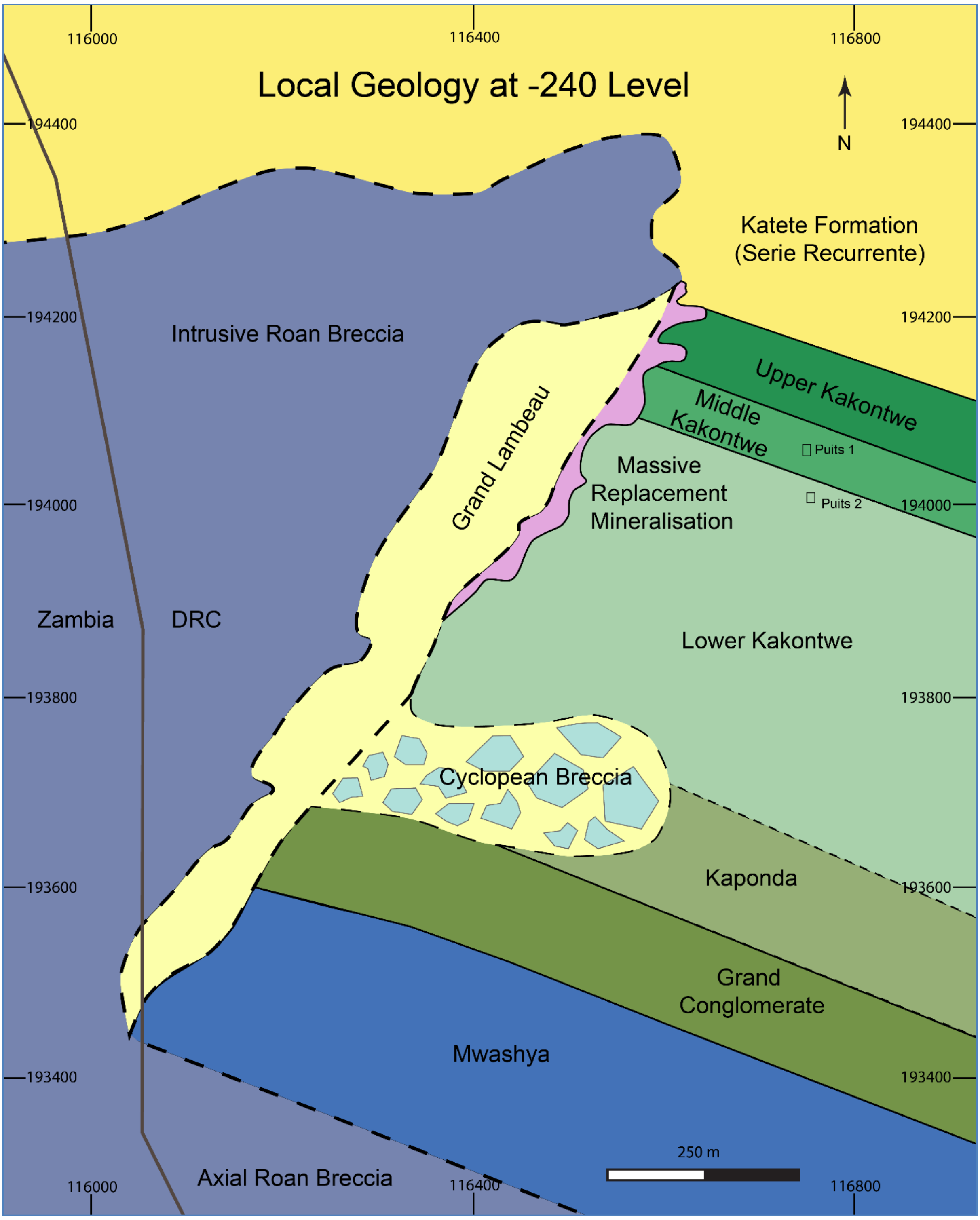
The Lower Kakontwe (LK) is a massive pale grey unit that consists almost entirely of microbial limestone. A range of variably preserved muddy-laminated (irregularly undulatory) to very distinctive calcimicrobial (laminated and clotted) lithofacies is present, with lesser volumes of featureless carbonate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

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

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

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

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



Source: Ivanhoe Mines (2018) adapted after Briart (1947).

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

7.2.2 Kipushi Reef Edge Interpretation

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

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

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

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

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

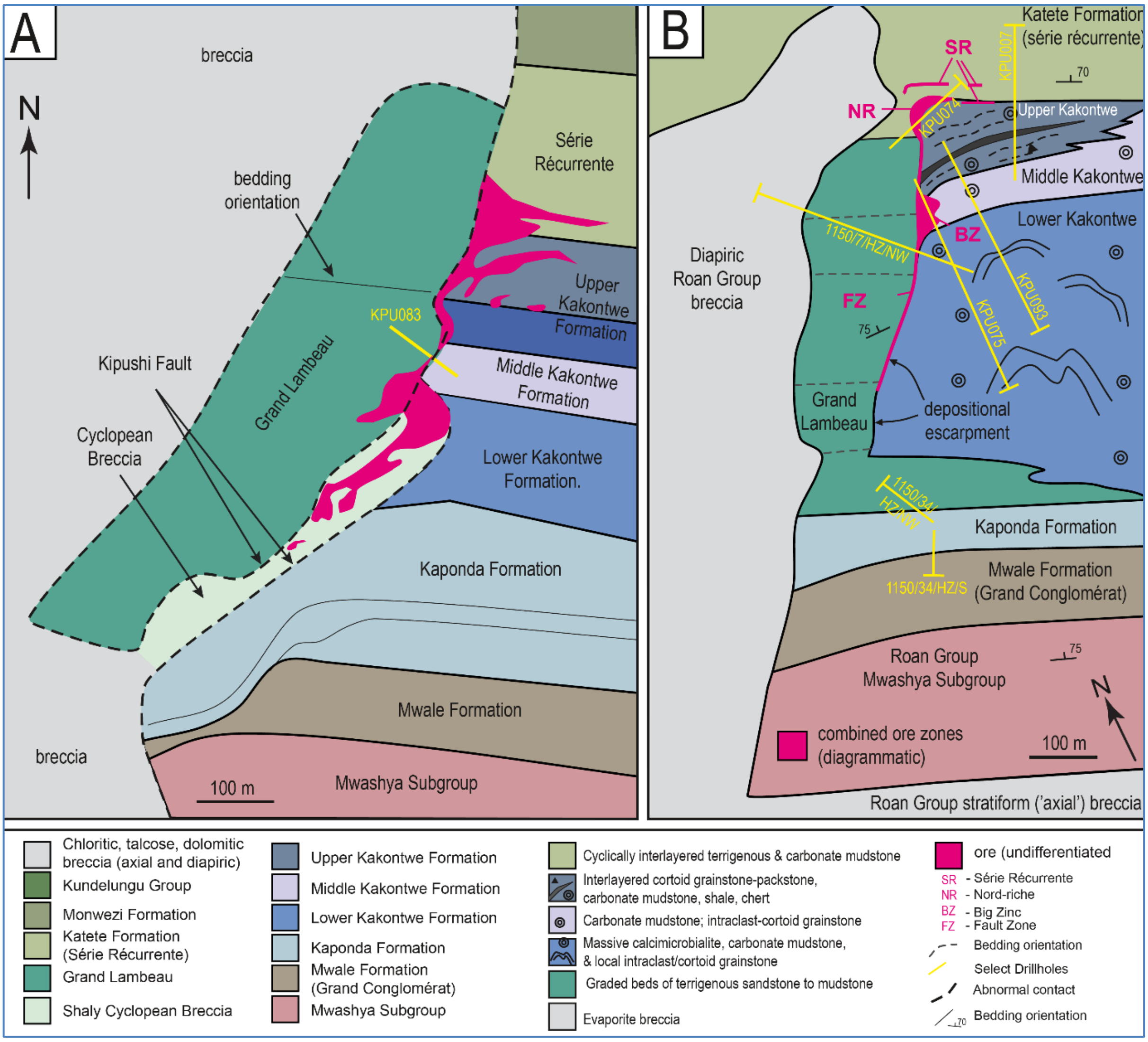
Various factor provide evidence to support this idea; (1) the fact that the Grand Lambeau, and the strata overlying the Série Récurrente are identical (Turner et al., 2018). (2) The bedding in the Kakontwe dolomites and the Grand Lambeau siltstone and sandstone units are roughly coplanar (Briart, 1947,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

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

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

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

Figure 7.8 Level Plan Showing Historical Interpretation (A) Compared to the Updated Reef Edge Interpretation (B) for the Kipushi Fault Zone (FZ)



Source: Ivanhoe Mines (2018) adapted after Turner et al (2018).

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

Table 7.1 Updated Stratigraphic Column for the Kipushi Project

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

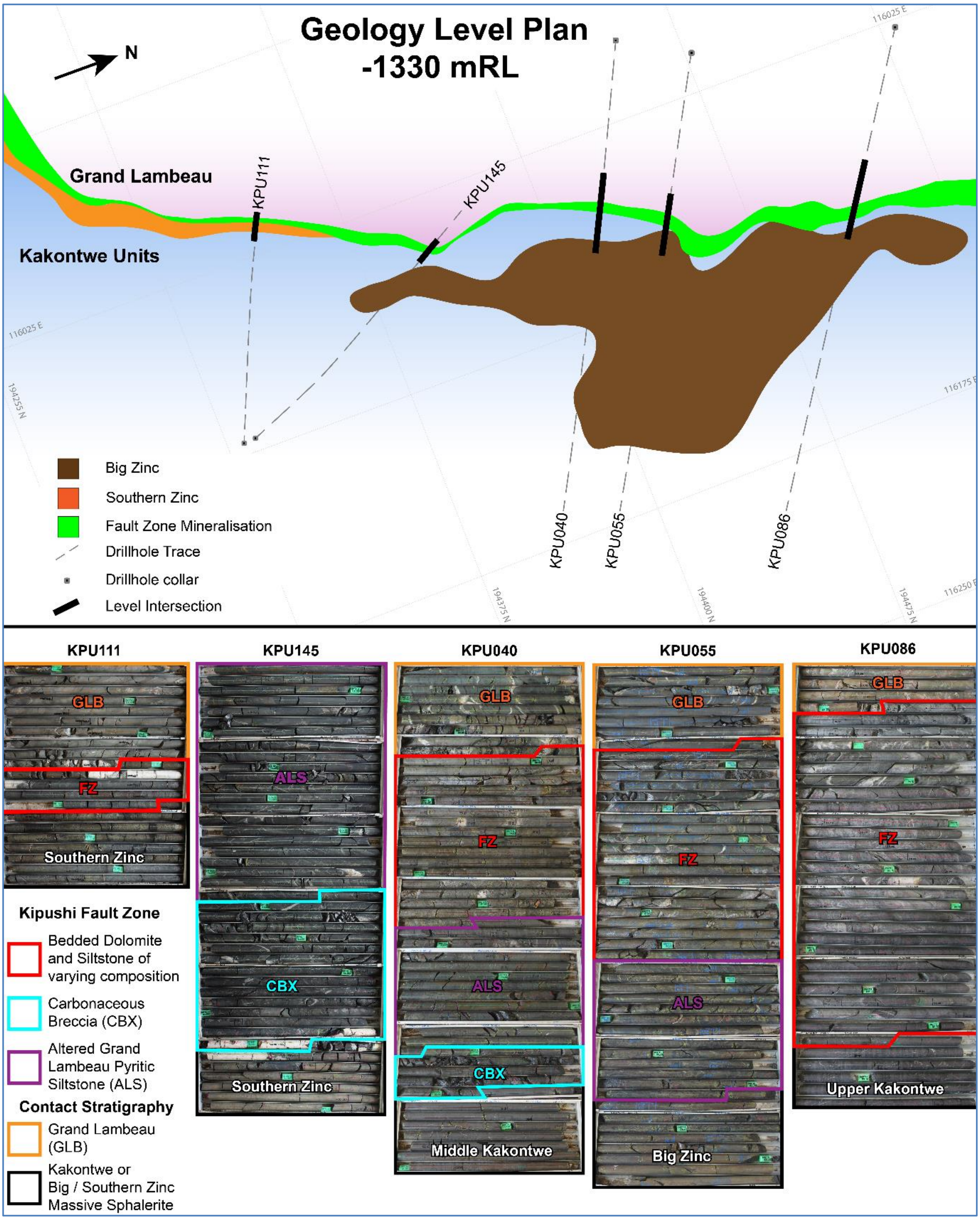
Reef Stratigraphy						
Subgroup		**Formation**		**Description**	**Traditional Congolese Designation**	**Mineralisation**
	Kaponda	Kaponda Formation		Finely laminated blue-grey to dark grey, sometimes cherty and carbonaceous dolostone, calcareous in places. Dark, tortuous, lenticular cherty and dolomicritic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

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

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

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

Figure 7.9 Lithological Variation Across the Kipushi Fault Zone at the -1330 mRL Level.



Source: Ivanhoe (2018). Note the intersections shown in the core photographs in the bottom half of the image are the drilled intersection and not true thickness.

Figure 7.10 Structural, and Mineralisation Variability Within the Kipushi Fault Zone (FZ)

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

7.3 Alteration and Metamorphism

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

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

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

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

7.4 Mineralisation

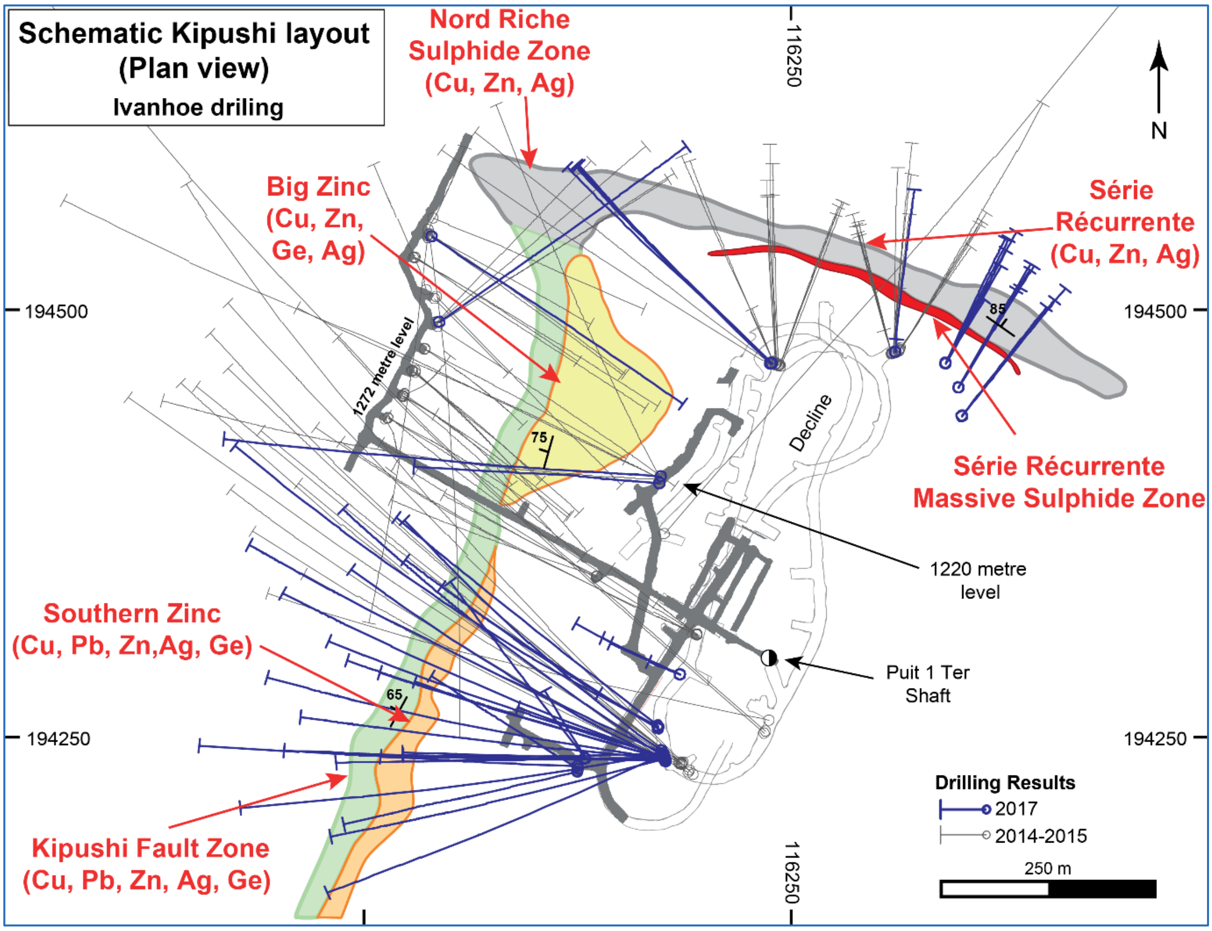
7.4.1 Overview

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

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

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

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

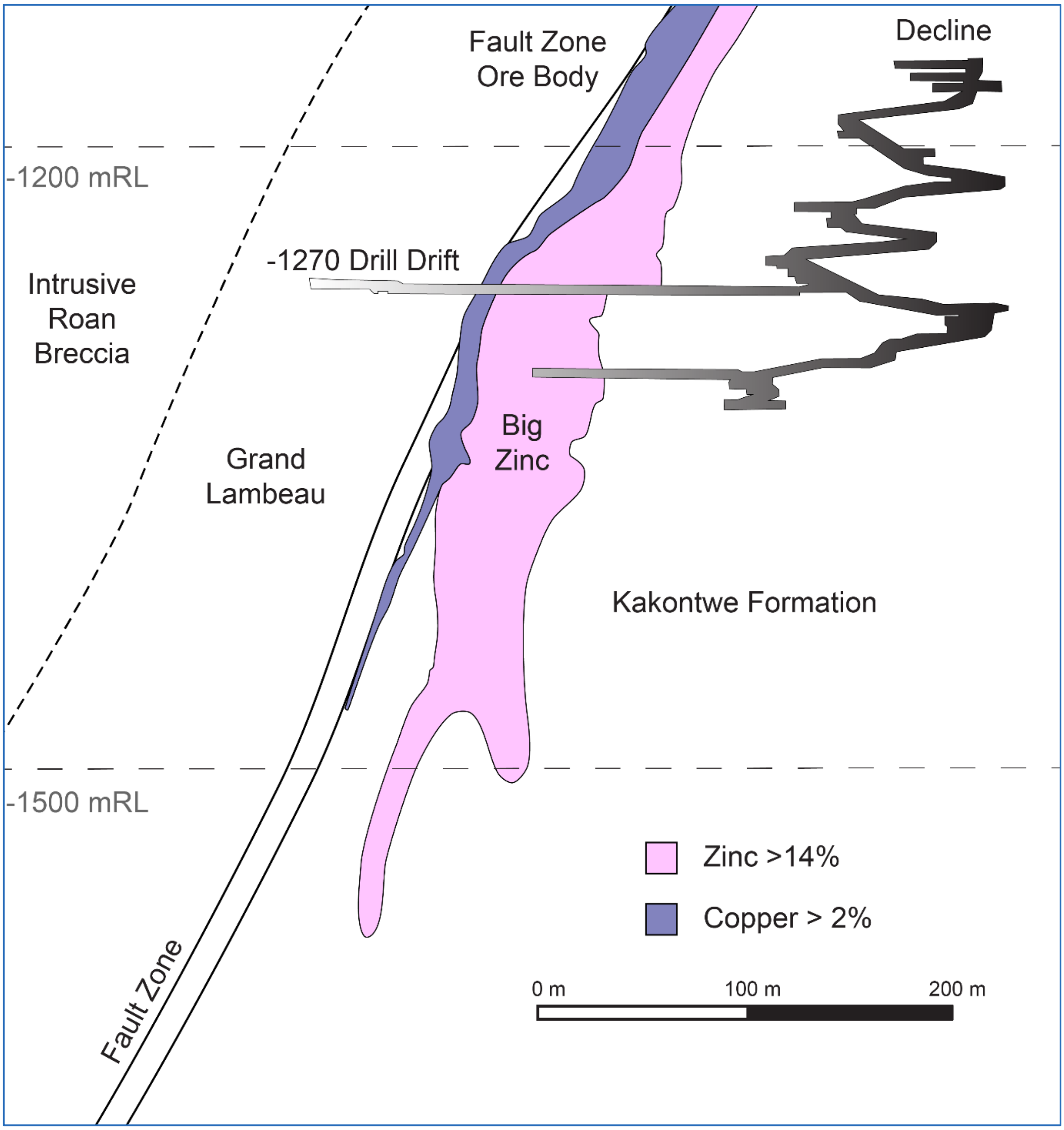


Source: Ivanhoe Mines (2018).

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

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

Figure 7.12 Cross-section Perpendicular to the Kipushi Fault, Looking North–north-east



Source: Ivanhoe Mines (2018).

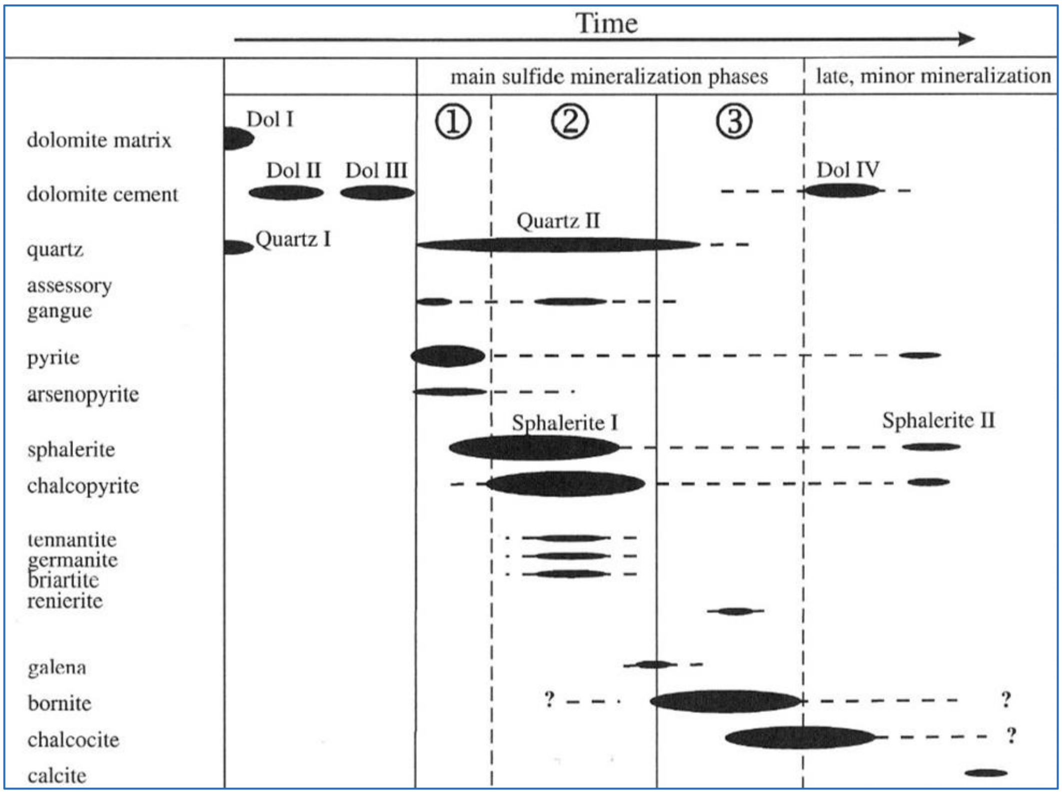
There is considerable variability in the mineralised zones, with a diverse range of economically significant accessory minerals for which Kipushi is well known. The complex mineralogy of Kipushi has been documented for over 60 years, although the lower levels of the deposit considered in this Kipushi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

Previous studies on the Kipushi mineralisation have shown that the sulphide mineralisation is complex and multiphase (e.g. Heijlen et al., 2008), and different generations of hydrothermal dolomite are also observed. A generalised paragenesis based on previous studies including work by Heijlen et al., (2008) is shown in Figure 7.13**Error! Reference source not found.**. As a typical feature, mineralisation formed through massive replacement of the dolomite host rock and cements, commonly resulting in banded mineralisation. Open space filling also occurred, but to a relatively minor extent. An initial sulphide phase of pyrite-arsenopyrite mineralisation 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 mineralisation in the higher levels of the mine.

There is a clear sulphide zonation from copper-rich at the Fault Zone contact, to zinc-rich, to zinc- and pyrite-rich massive sulphide at the contact with the Kakontwe Formation (left to right in Figure 7.13). This mineral zonation is similar to that seen in other Central African Copperbelt deposits, wherein copper is proximal to source (for example, the FZ) whereas zinc and pyrite are distal. Lead appears to be controlled, at least partially, by the Kakontwe stratigraphy, with the highest lead grades corresponding with the Upper – Middle Kakontwe contact.

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

Figure 7.13 Generalised Paragenesis of Mineralisation and Gangue at Kipushi

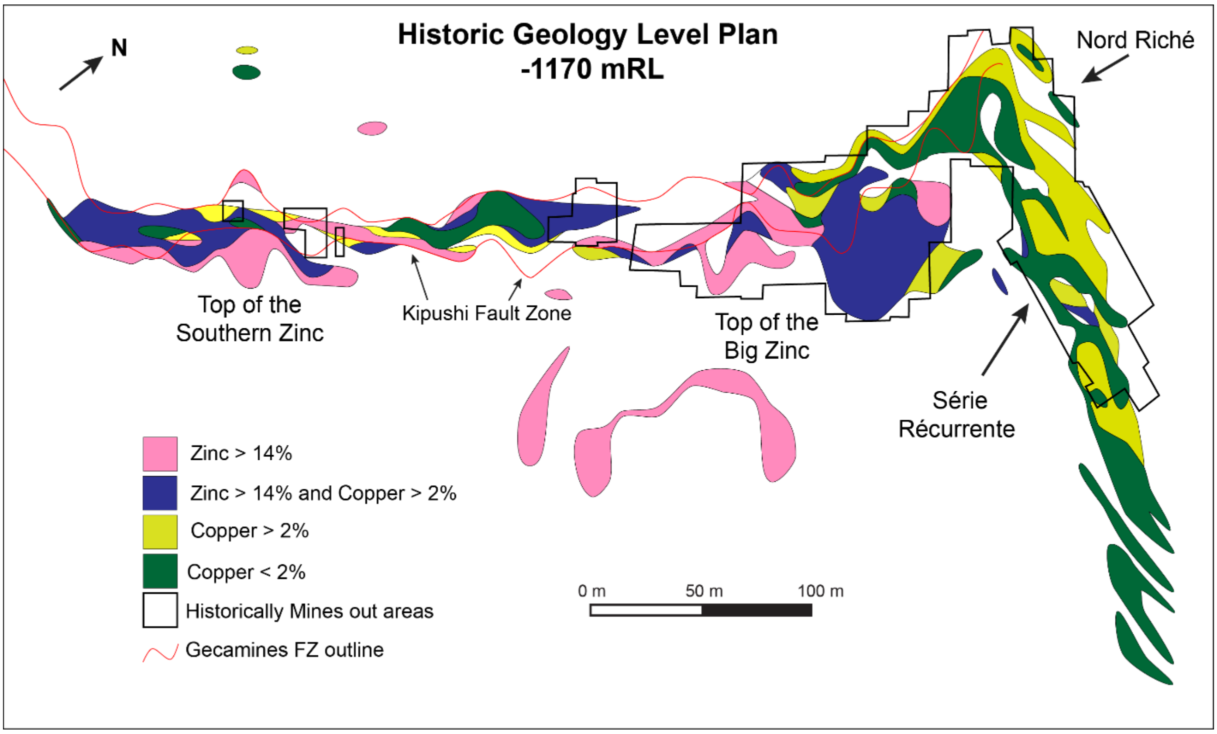


Source: Heijlen et al., (2008).

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

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

Figure 7.14 Historic Level Plan from -1170 mRL, Showing the Different Historic Mining Zones



Source: Ivanhoe mines (2018).

7.4.2 The Big Zinc

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

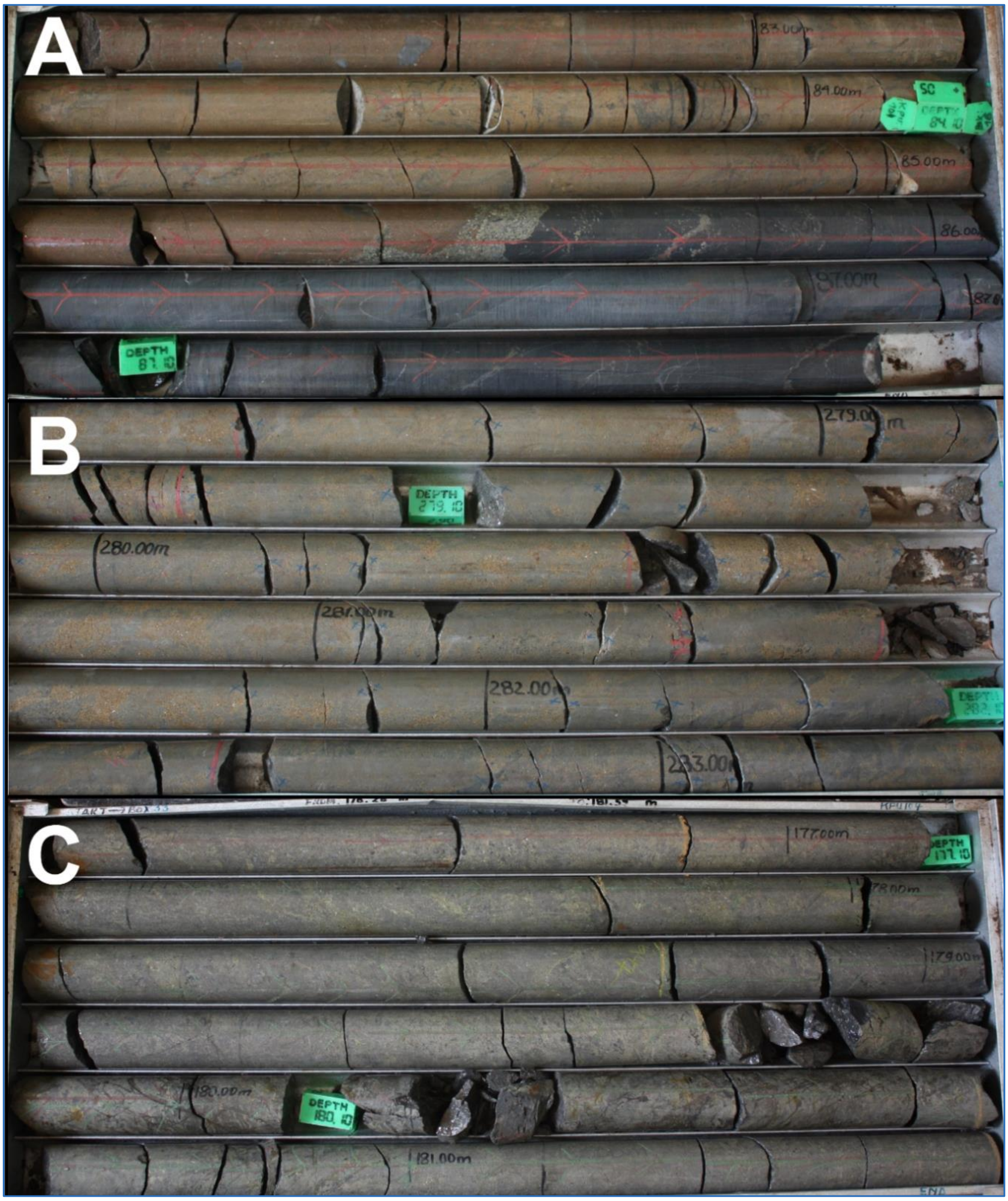
The contacts of mineralisation with the host Kakontwe dolomite are zoned over several metres. Sphalerite on the margins of the mineralised zone, particularly at the terminations of apophyses, is often red and iron-rich (Figure 7.15) and associated with arsenopyrite, and commonly grades outwards to a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

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

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

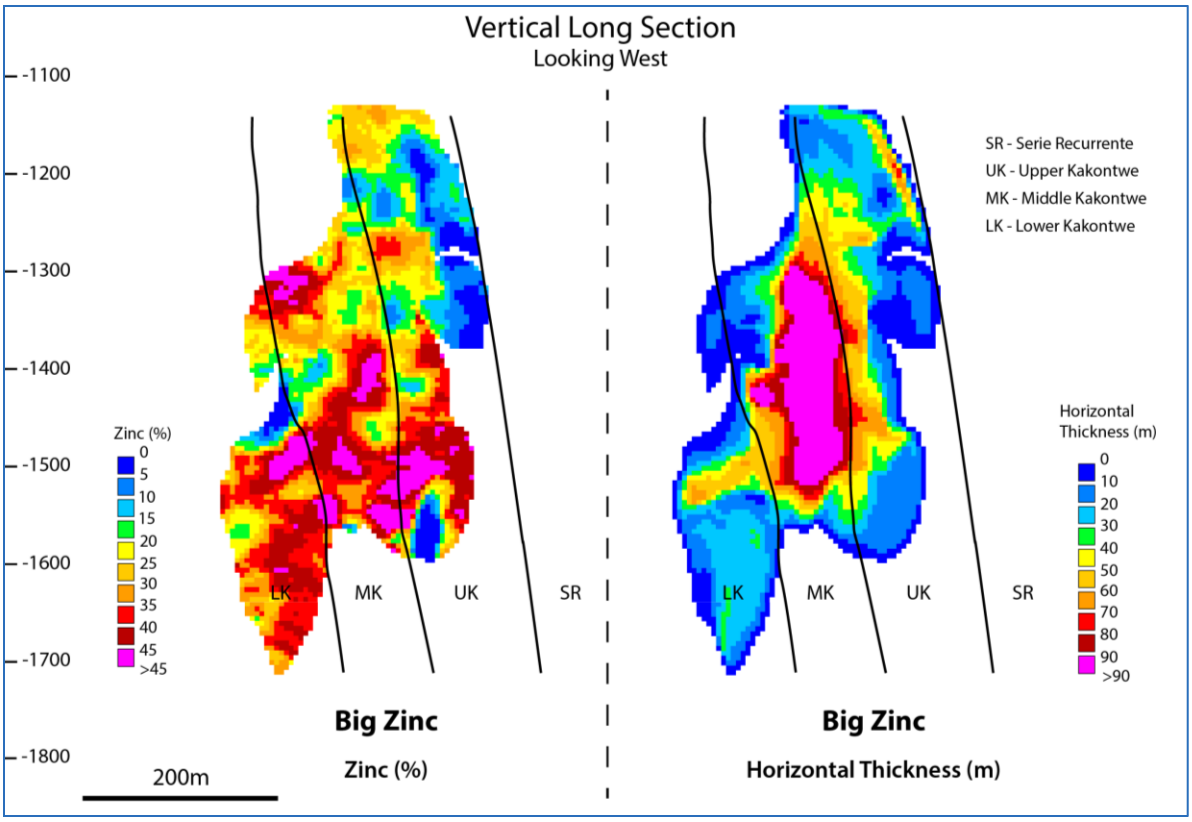
Localised internal barren to lower-grade "stérile" zones occur and were defined by Gécamines on the visual basis of 7% Zn and/or 1% Cu cut-offs. Drill core from these zones was generally not preserved by Gécamines.

Figure 7.15 Mineralisation Intersected in Metallurgy Drillhole KPU104: A) Reddish iron-rich sphalerite and pyrite on the margins of the Big Zinc, B) the dominant massive honey-brown sphalerite, and C) Copper rich mineralisation (chalcopyrite, tennantite and silver sphalerite) of the Cu-Zn zone in the central part of the Big Zinc



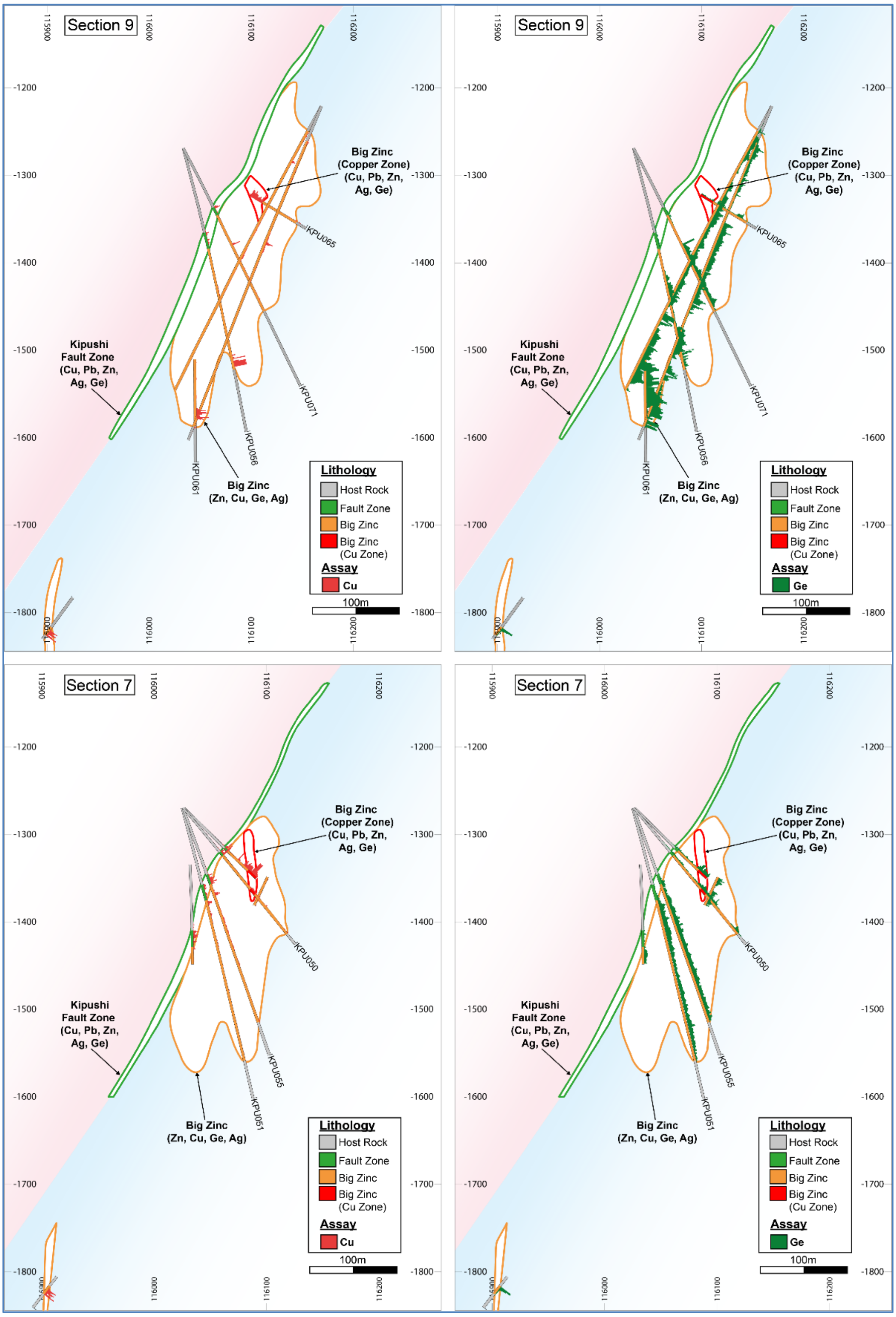
Source: Ivanhoe Mines, 2018.

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



Source: Ivanhoe Mines (2018).

Figure 7.17 Copper and Germanium distribution through the Big Zinc (Sections 5 and 9)



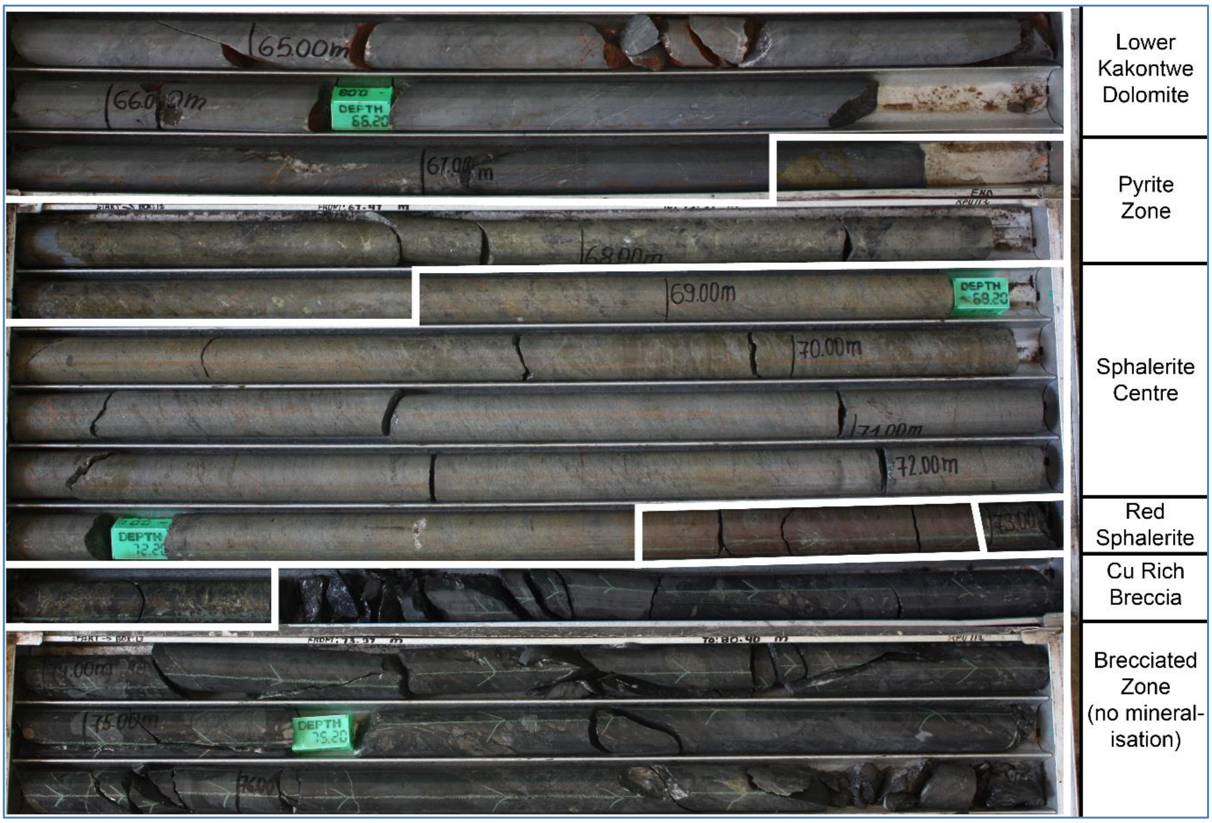
Source: Ivanhoe Mines (2018).

7.4.3 Southern Zinc

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

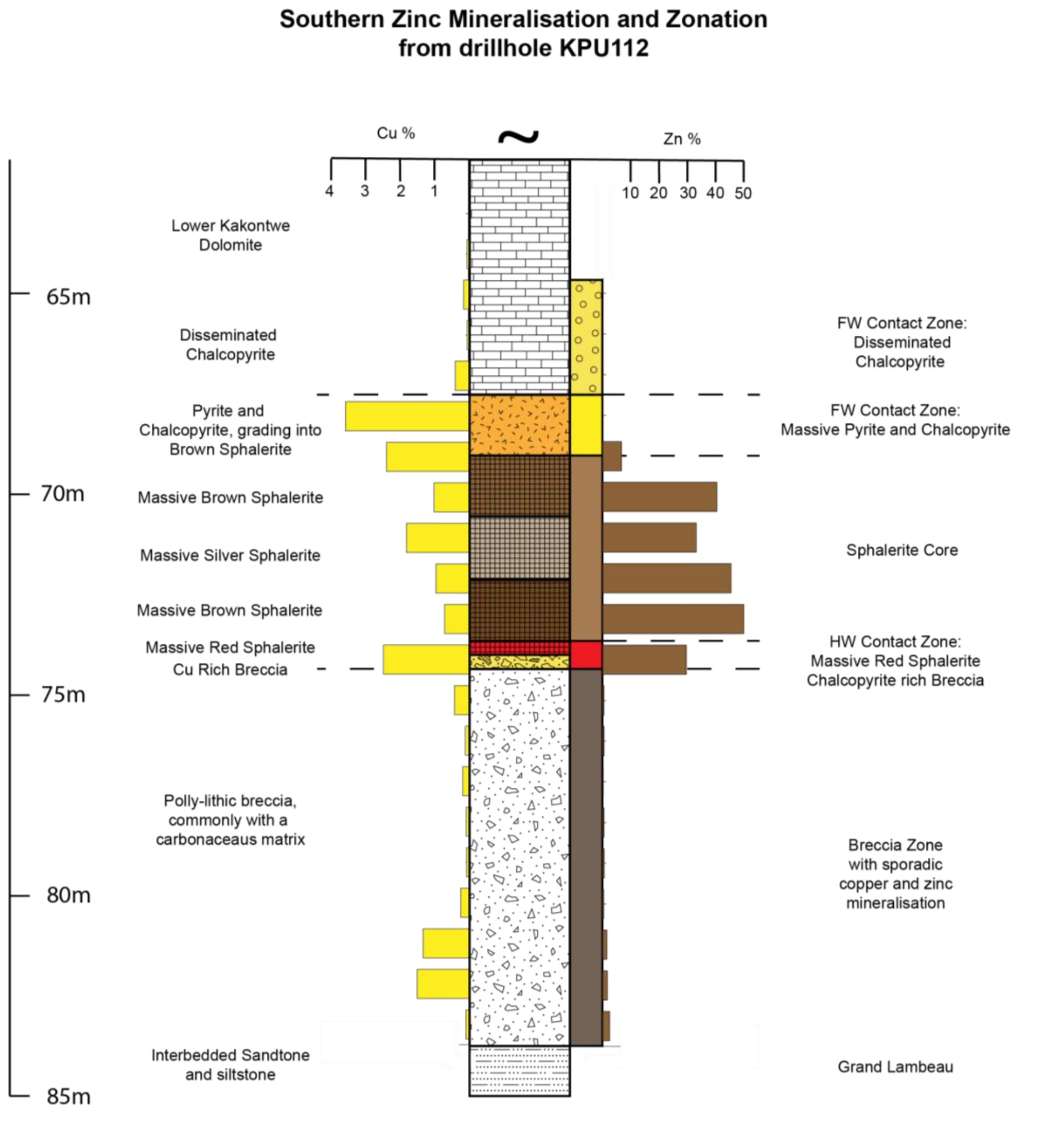
The Southern Zinc displays a definite mineralisation zonation. Disseminated chalcopyrite occurs within the footwall dolomites (not present in all the drill holes), followed by an abrupt change to massive sulphide mineralisation closer to the Fault Zone. The massive sulphides are zoned as follows; a pyrite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

Figure 7.18 Southern Zinc Mineralisation Zonation from Drillhole KPU112



Source: Ivanhoe Mines (2018).

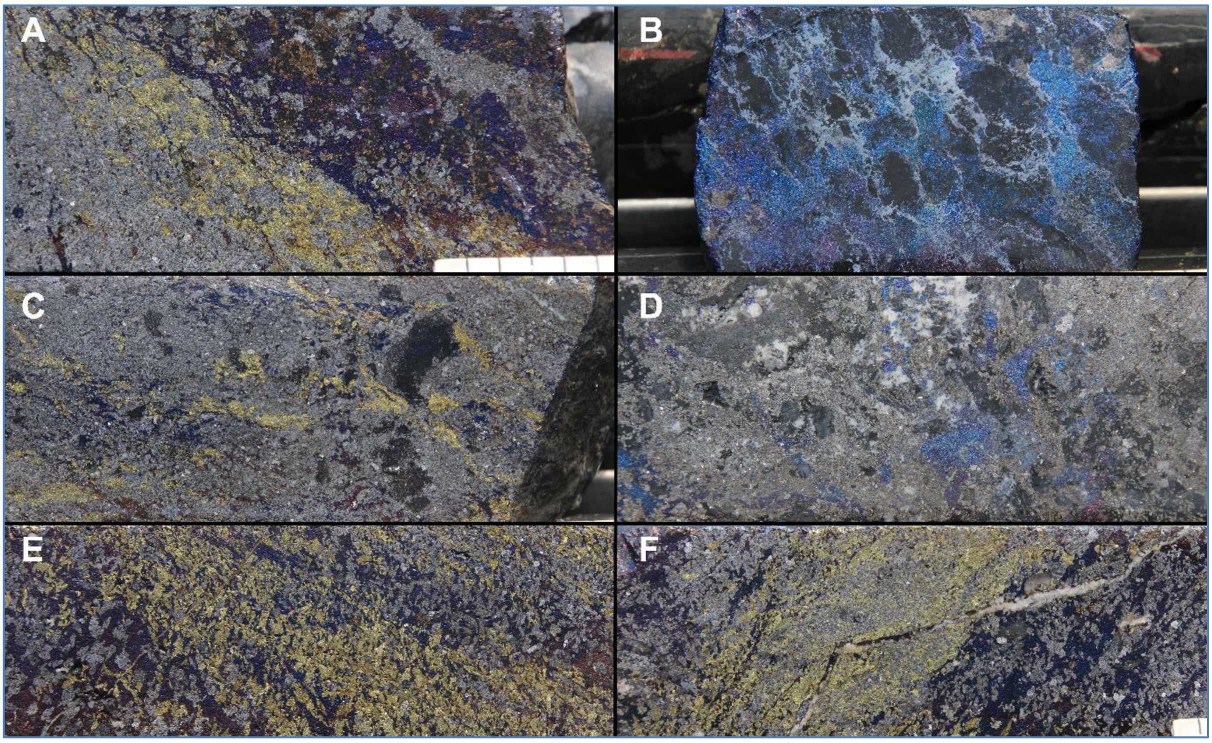
Figure 7.19 Idealised Southern Zinc Mineralisation Zonation



Source: Ivanhoe Mines (2018).

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

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



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

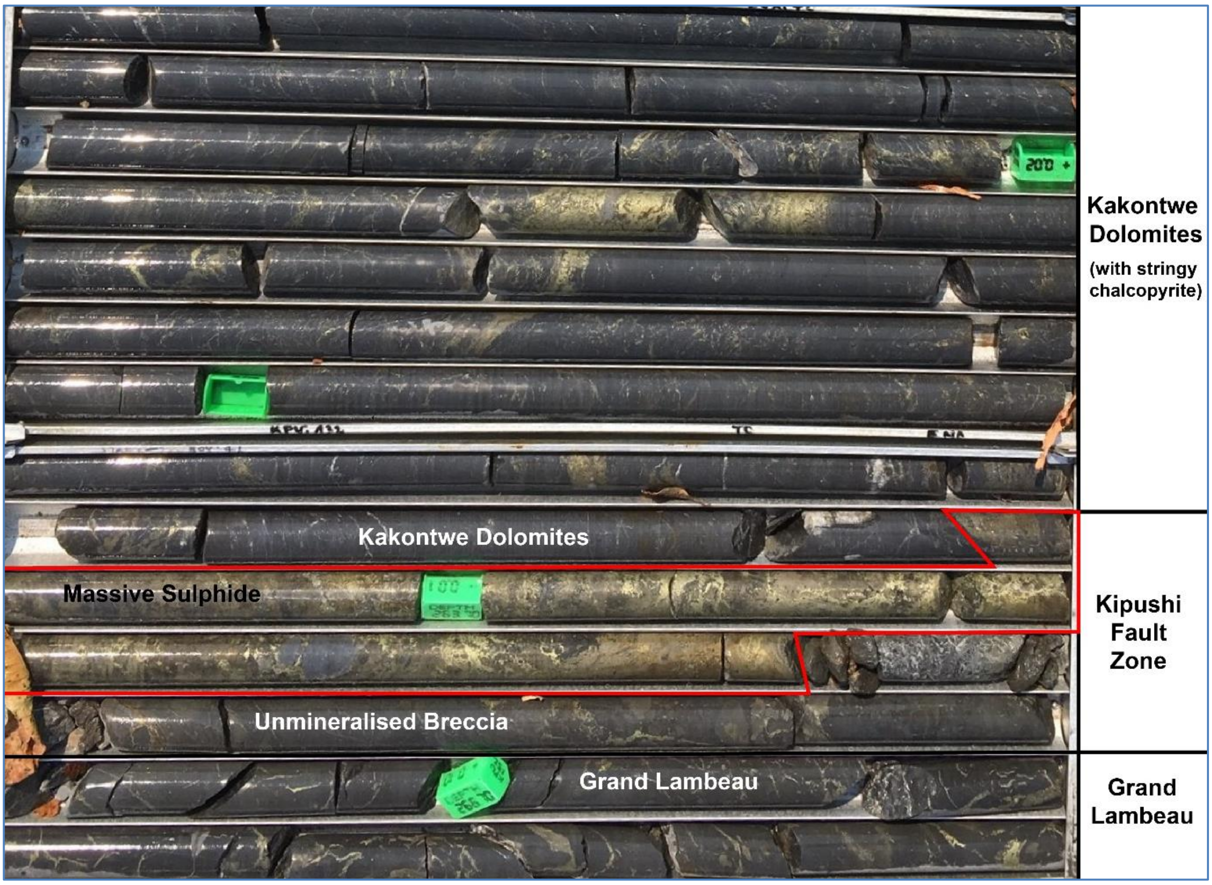
7.4.4 Kipushi Fault Zone

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

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

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

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



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

7.4.5 Copper Nord Riche

Discreet mineralised zones, of patchy to massive chalcopyrite with minor sphalerite are focussed at the top of the Upper Kakontwe Formation (Figure 7.21), near its contact with the Série Récurrente Formation. This area is locally known as the Copper Nord Riche. Mineralisation in the Copper Nord Riche is significantly thicker than in the adjacent Série Récurrente. In the Copper Nord Riche, the mineralised zones are oblate and discordant, cutting down stratigraphy and thickening in closer proximity to the Kipushi Fault Zone, especially at the termination of the Upper Kakontwe against the Fault Zone (Figure 7.11 and Figure 7.23). Chalcopyrite intercepts frequently contain elevated silver (>100 ppm), arsenic (>5000 ppm) and molybdenum (>100 ppm), associated with tennantite (Figure 7.23).

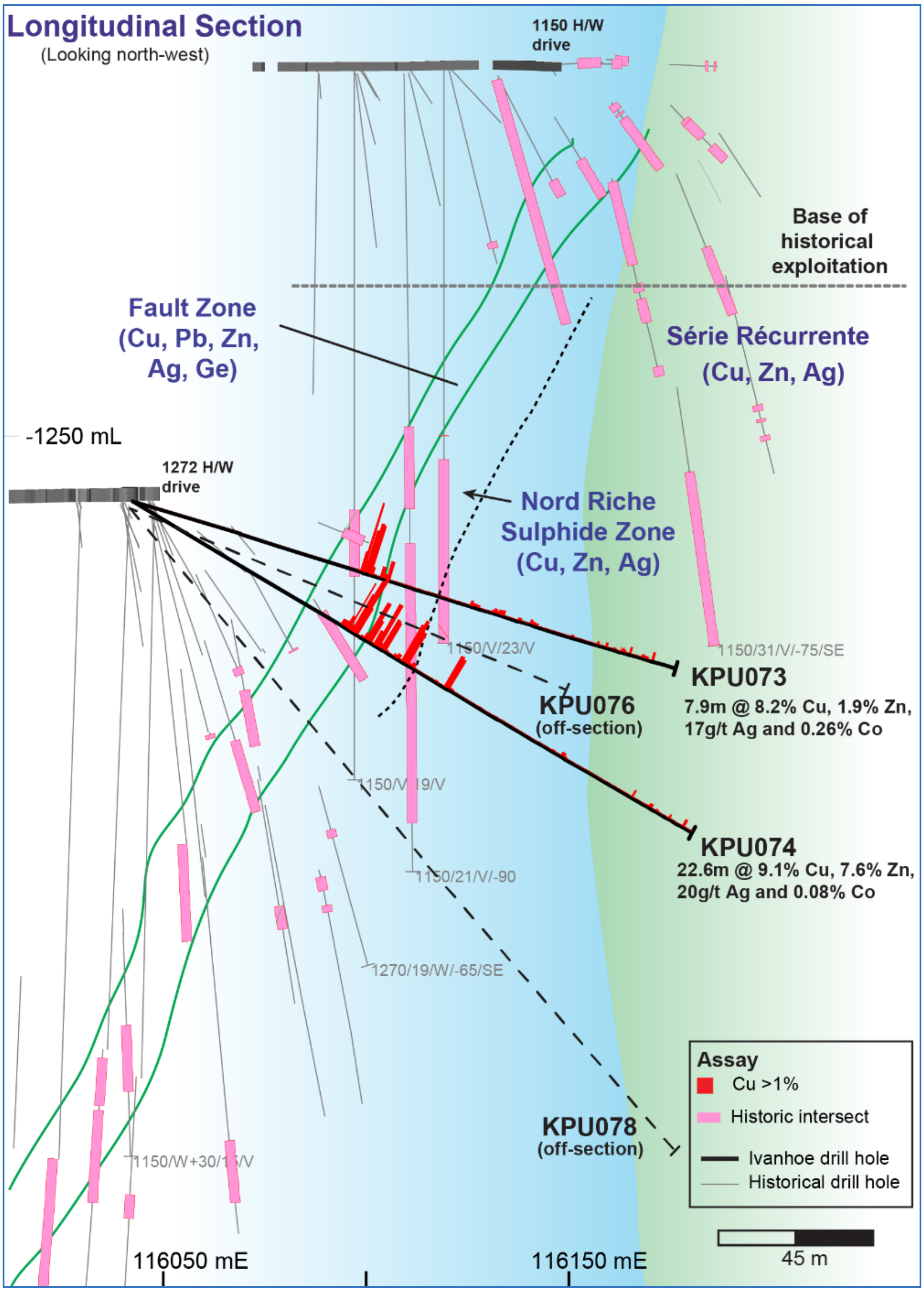
Replacement mineralisation in the Upper Kakontwe has an association with locally disrupted bedding. Parasitic folds in the plane of bedding, plunging at steep angles, seem to localise mineralisation and replacement.

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

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

Source: Ivanhoe Mines (2018).

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



Source: Ivanhoe Mines (2018).

7.4.6 Série Récurrente

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

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

7.4.7 Disseminated Série Récurrente Mineralisation

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

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

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

Source: Ivanhoe Mines (2015).

Figure 7.25 Blebby and Disseminated Chalcopyrite in Série Récurrente Siltstone at 148 m in Drillhole KPU074. Both Mineralisation and Bedding are Deformed by Parasitic Folds

Source: Ivanhoe Mines (2015).

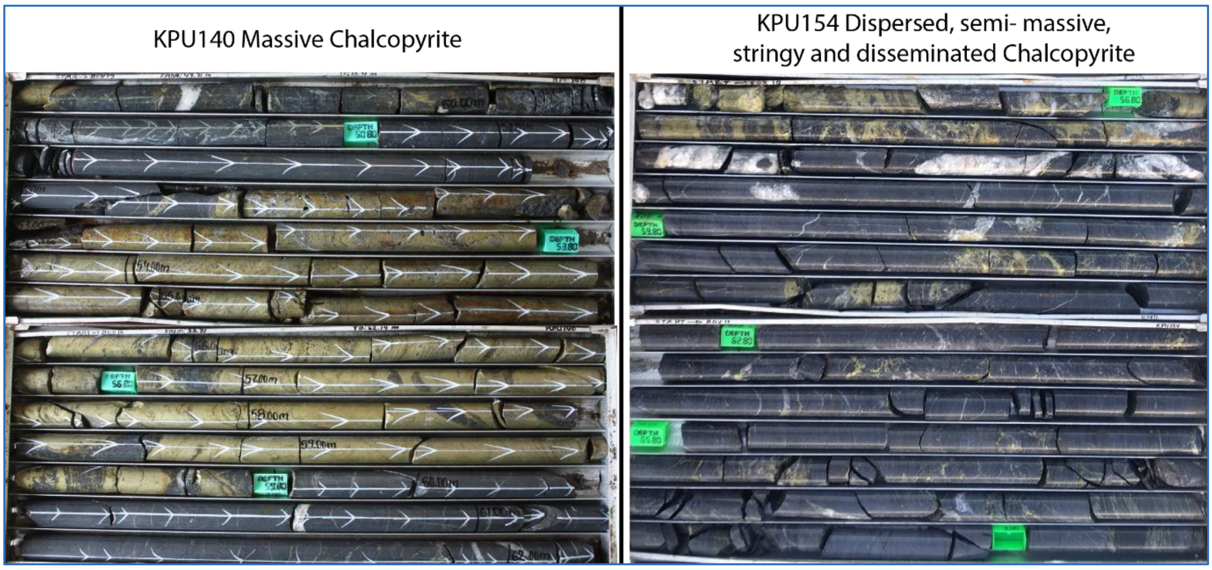
7.4.8 Massive Sulphide Pod

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

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

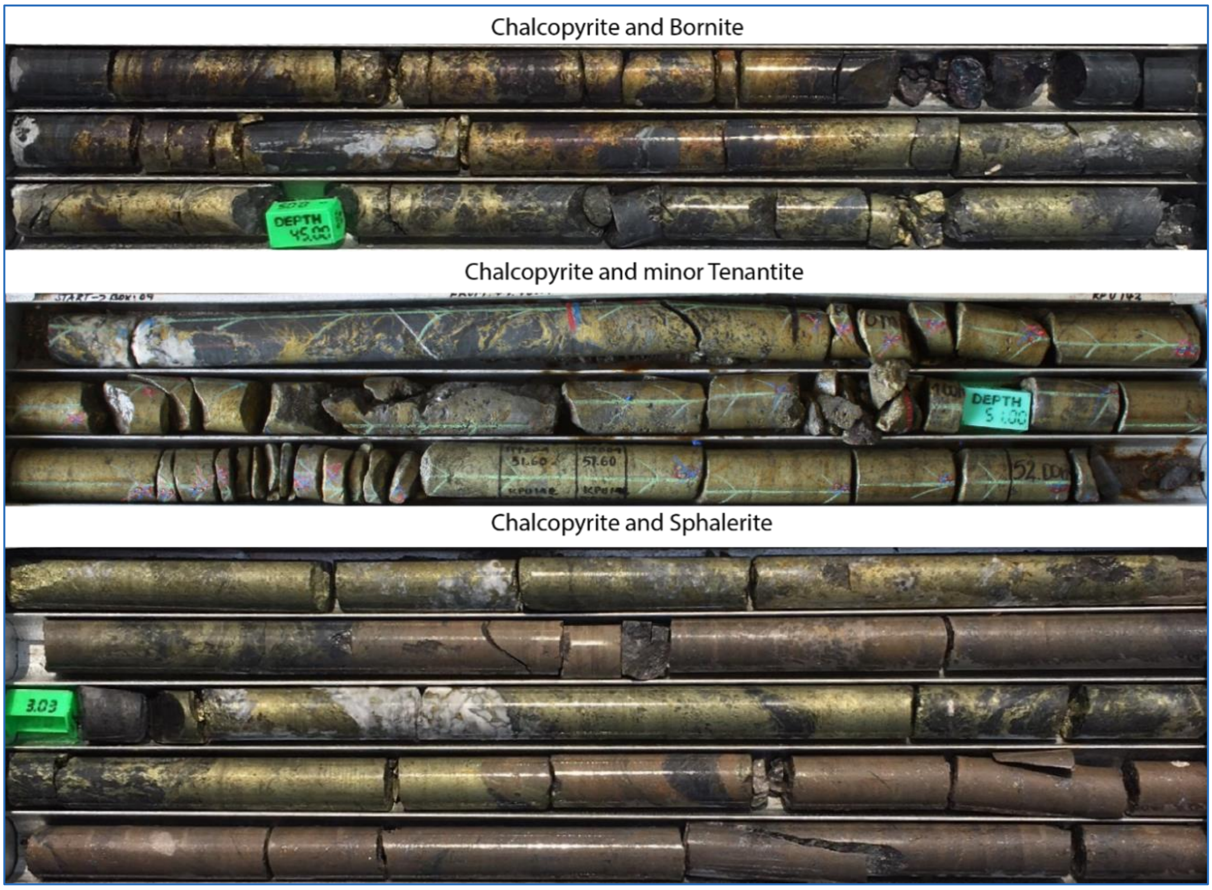
There is also a distinct mineral zonation evident within the sulphide body, chalcopyrite dominates, but there is a clear vertical zonation; a mixture of chalcopyrite and bornite up-plunge; a central portion of mostly chalcopyrite with some minor tennantite; and down-plunge, a mixture of chalcopyrite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

Figure 7.26 Mineralisation Style within the High-Grade Pod



Source: Ivanhoe Mines (2018).

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



Source: Ivanhoe Mines (2018).

8 DEPOSIT TYPES

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

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

No association with igneous rocks is observed. Mineralisation forms as fault or breccia filling, and massive replacement mineralisation with either abundant diagenetic pyrite or other source of 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-mineralisation plumbing systems were typically created by karsting, faulting, collapse zones as a result of evaporate removal, and/or bedding plane aquifers and were enhanced by volume reduction during 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, as well as lenticular bodies subparallel to stratigraphy. They are thought to be associated with collapse breccias and faults (Kampunzu, et al., 2009 and tend to be surrounded by silicified dolomite. These carbonate-hosted copper-zinc-lead deposits tend to contain important by-products of silver, cadmium, vanadium, germanium, and gallium.

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

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

9 EXPLORATION

No other relevant exploration work, other than drilling, has been carried out by KICO on the Kipushi 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 and 12.5 m apart along the Série Récurrente, with each section consisting of a fan of between four and seven holes (Figure 10.1), the angle between the holes being approximately 15° (Kelly et al., 2012). Sections are even-numbered south of Section 0 and odd-numbered to the north. Drilling was completed along the Kipushi Fault Zone from Section 0 to 19 along a 285 m strike length including a 100–130 m strike length which also tested the Big Zinc and a 50–200 m interval that tested the Southern Zinc.

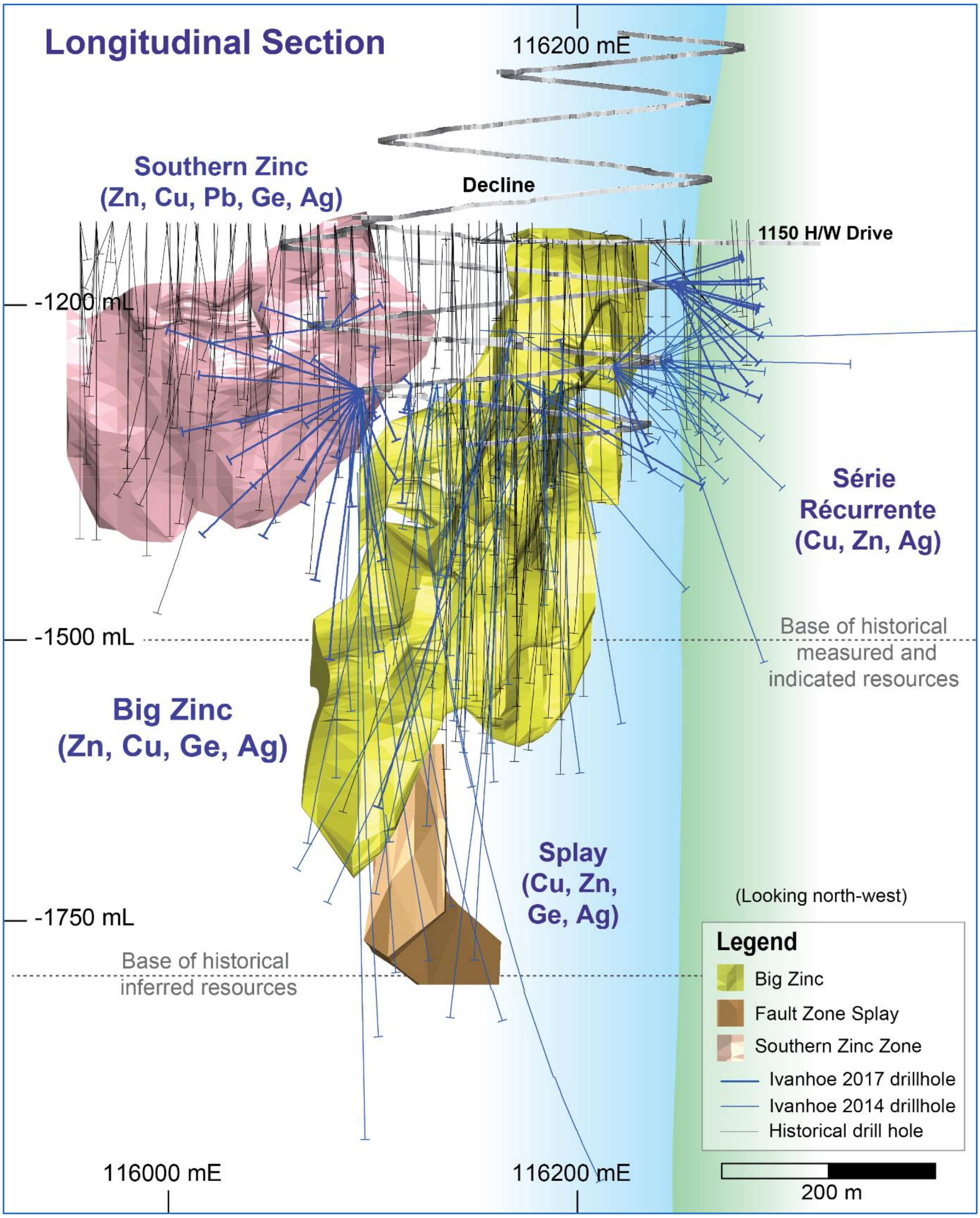
Drill core from 49 of the 60 holes drilled from 1,272 mRL that intersected the Big Zinc are stored under cover at the Kipushi mine. The retained half core is in a generally good condition and is mostly BQ in size with subordinate NQ core. In general, only mineralised intersections were retained, with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

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

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

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

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



Source: Ivanhoe Mines (2018).

10.1.2 Drillhole Database

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

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

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

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

10.2 KICO Drilling

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

10.2.1 Drilling Methodology

The Kipushi mine was placed on care and maintenance in 1993 and flooded in early 2011 due to a lack of pumping maintenance over an extended period. Following dewatering and access to the main working level in December 2013, an original 25,400 m underground drilling programme was carried out by KICO between March 2014 and October 2015 A subsequent 9,700 m drilling campaign was carried out from May to October 2017. 24 April 2018 is the cut-off date for data included in the Mineral Resource.

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

- Conduct confirmatory drilling to validate the Historical Estimate within Kipushi's Big Zinc and Fault Zone and qualify them as current Mineral Resources prepared in conformance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) standards as required by National Instrument 43-101.
- Conduct extension drilling to test the deeper portions of the Big Zinc and Fault Zone below 1,500 mRL.
- Test for deeper extensions to the Big Zinc 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 mineralisation of the mine's Copper Nord Riche.
- Gain an improved understanding of geology and controls on mineralisation.

The 2017 drilling campaign focussed on the following:

- Metallurgical and geotechnical drilling, to be used in the testwork for Section 13 and Section 16 respectively (Table 10.2).
- Confirm and expand the Southern Zinc, with the aim to upgrade the Inferred Mineral Resource into an Indicated Mineral Resource.
- To better understand the link between the Big Zinc and Southern Zinc mineralisation.
- Expansion drilling in the Série Récurrente, further delineating the high-grade pod.
- Target the Nord Riche and investigate the potential mineralisation close to the 1272 hangingwall drive.

Underground drilling of the various mineralised 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 were used again in the 2017 drilling campaign operating two Boart Longyear LM90 electro-hydraulic underground drill rigs.

Drilling was carried out on the same 15 m spaced sections used by Gécamines and comprised twin holes, infill holes and step-out exploration holes. Drilling on each section comprised a fan of between four and seven holes. The angle between the holes was +/- 15°. KICO drilled the different targets from different locations. The northern part of the Big Zinc was drilled from the 1,272 mRL hanging-wall drill drive along the Fault Zone. The southern portion of the Big Zinc and the Southern Zinc was drilled from the decline below 1272 mRL, while the Série Récurrente drilling was performed from two levels in the decline, ~1180 mRL and ~1250 mRL.

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

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

Table 10.1 Underground Drilling Summary

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

Table 10.2 2017 Metallurgy and Geotechnical Drilling Program Breakdown

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

10.2.2 Core Handling

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

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

10.2.3 Core Recovery

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

Core recovery averaged 99.14% and visual inspection by the 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, mineralisation, and structure were recorded on standardised paper templates and then captured and validated on import into the MS Access database. From hole KPU015 onwards, all logging was done directly into MS Access. All geotechnical logging was done directly into MS Access.

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

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

10.2.6 Results

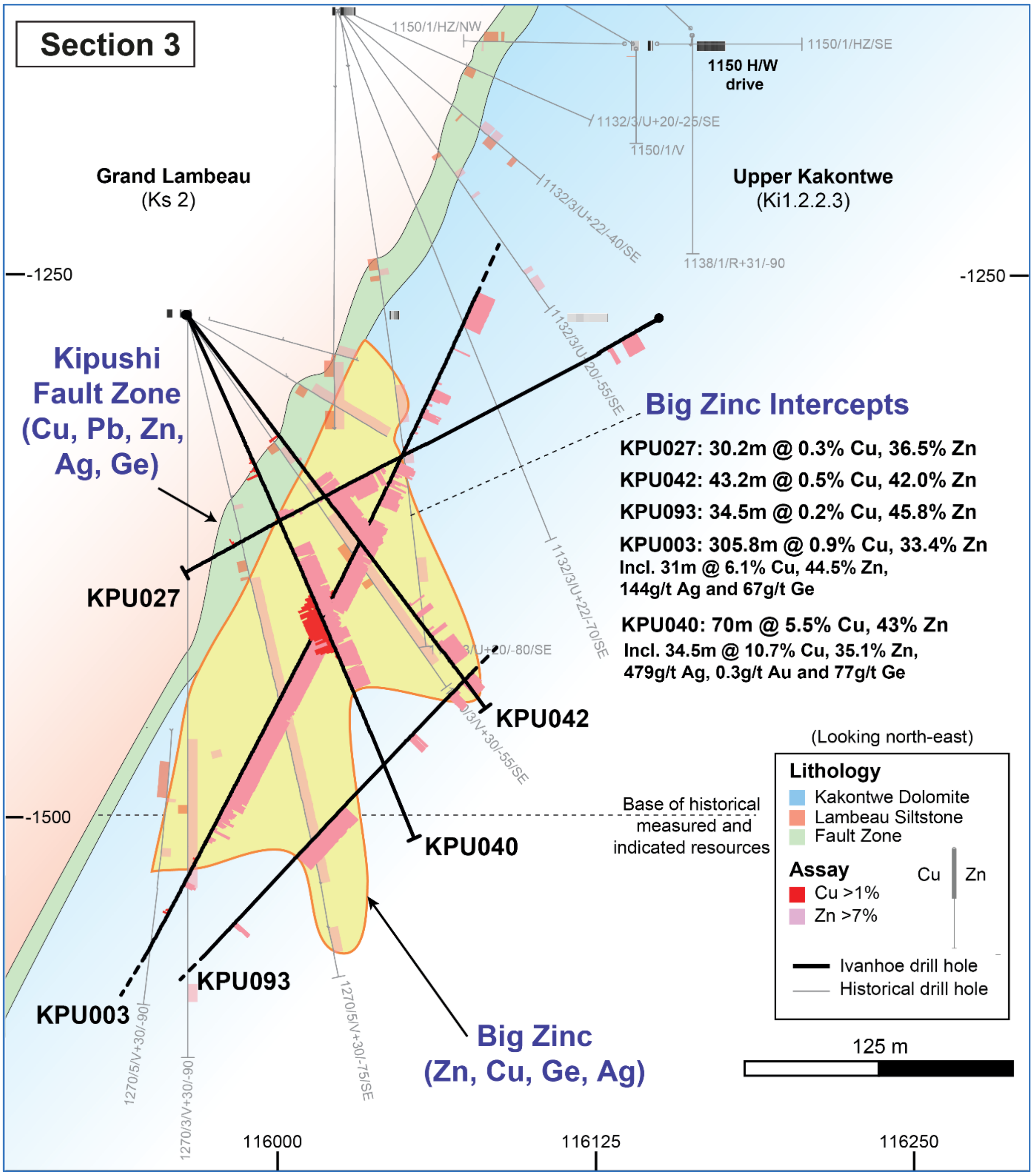
The KICO drilling focussed on the zinc dominant targets of the Big Zinc and Southern Zinc, as well as the copper dominant zones of the Fault Zone, the Nord Riche and the Série Récurrente. The deep extension of the Kipushi Fault Zone and the associated Splay have also intersected copper and zinc mineralisation.

10.2.6.1 Big Zinc

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

Figure 10.2 shows section 3, illustrating the KICO drilling results within the Big Zinc and Fault Zone. The geometry of the Big Zinc and copper-rich and zinc-rich mineralised zones at depth below the Big Zinc are shown in Figure 10.3.

Figure 10.2 Drill Section 3 Showing Drillholes through the Big Zinc

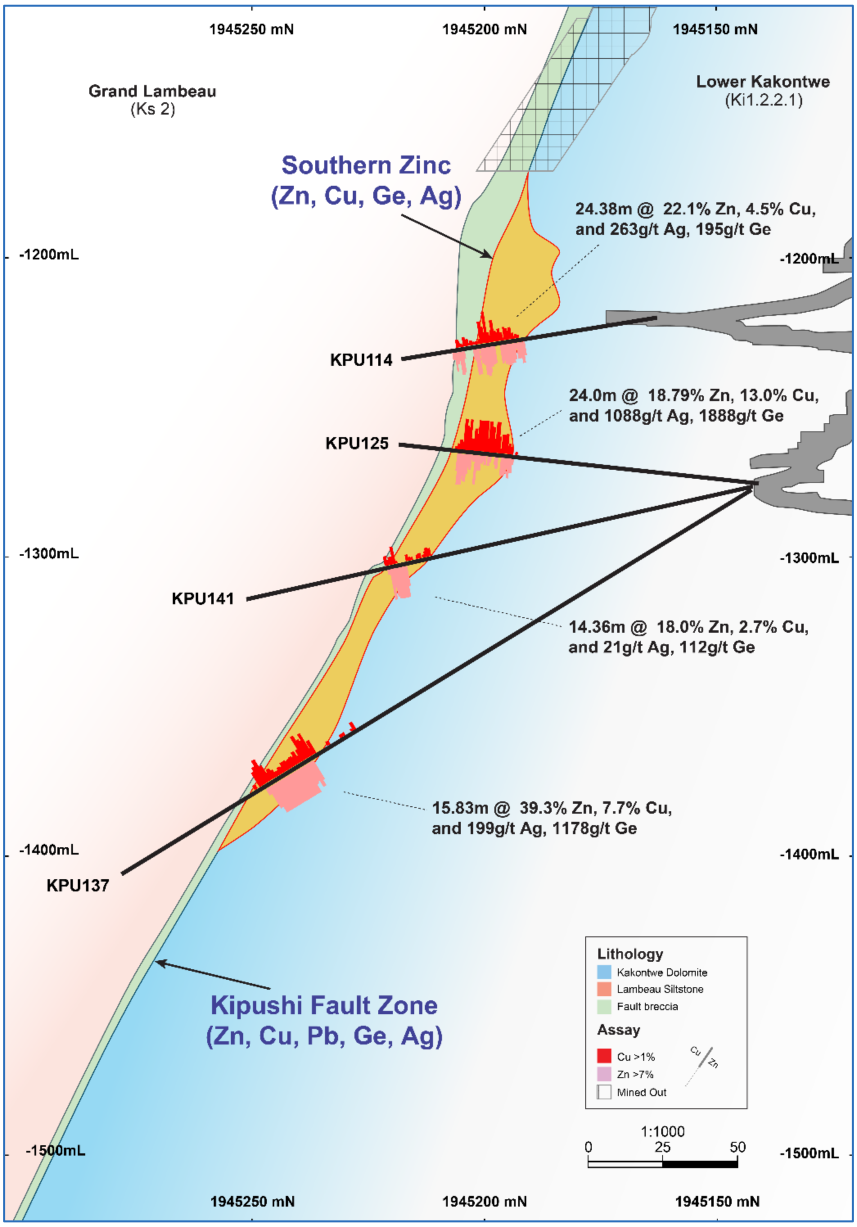


Source: Ivanhoe Mines (2018).

10.2.7 Southern Zinc

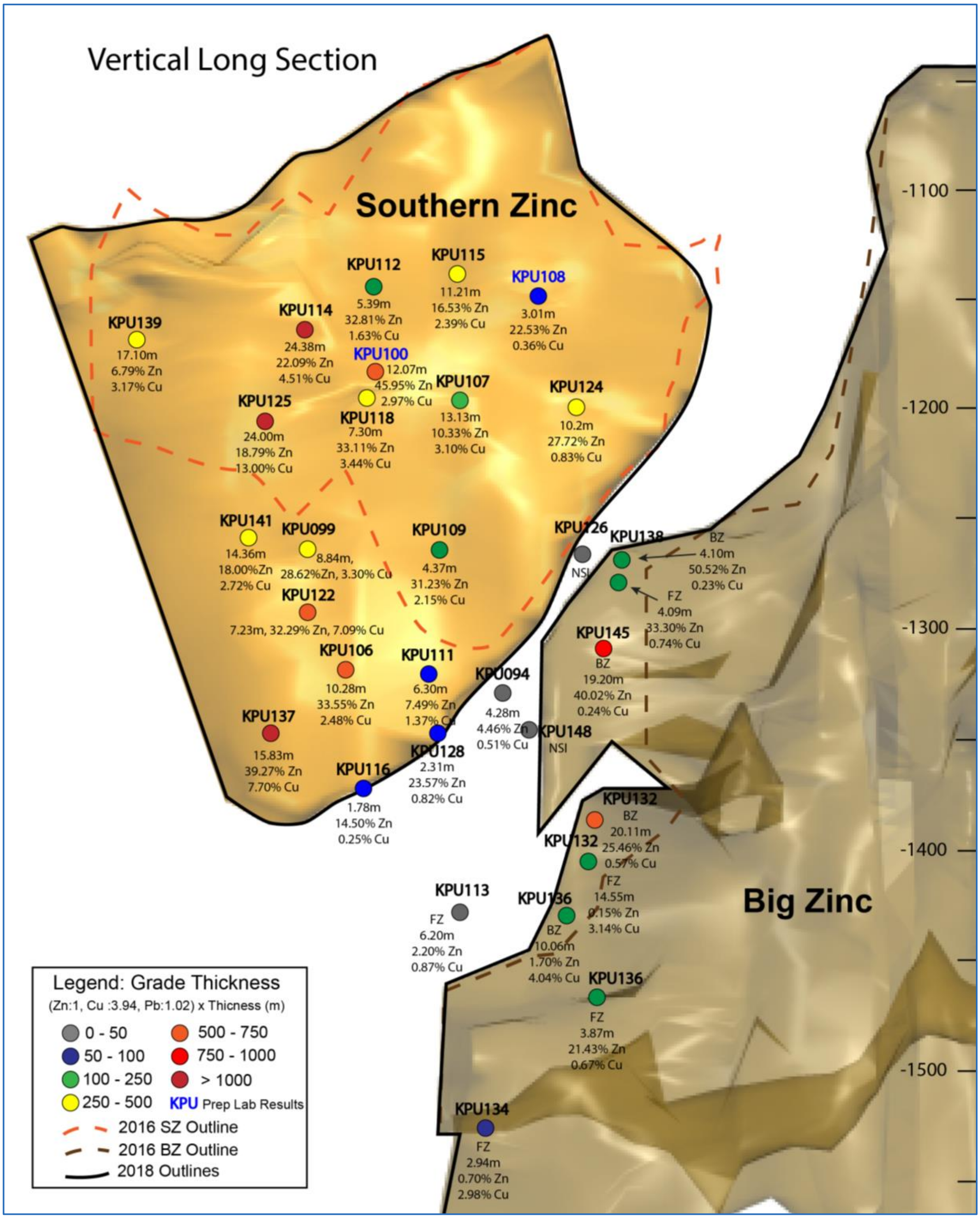
The polymetallic Southern Zinc is located south of the Big Zinc, abutting the Fault Zone (Figure 7.14). The zinc grade is lower when compared to the Big Zinc; while comparably, most of the other elements are elevated, including copper, silver, lead, arsenic, cadmium and germanium. This increase appears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

Figure 10.3 Southern Zinc Cross Section



Source: Ivanhoe (2018). The thickness are shown as true thickness.

Figure 10.4 Expanded Southern Zinc and Big Zinc Vertical Long Section



Source: Ivanhoe Mines (2018).

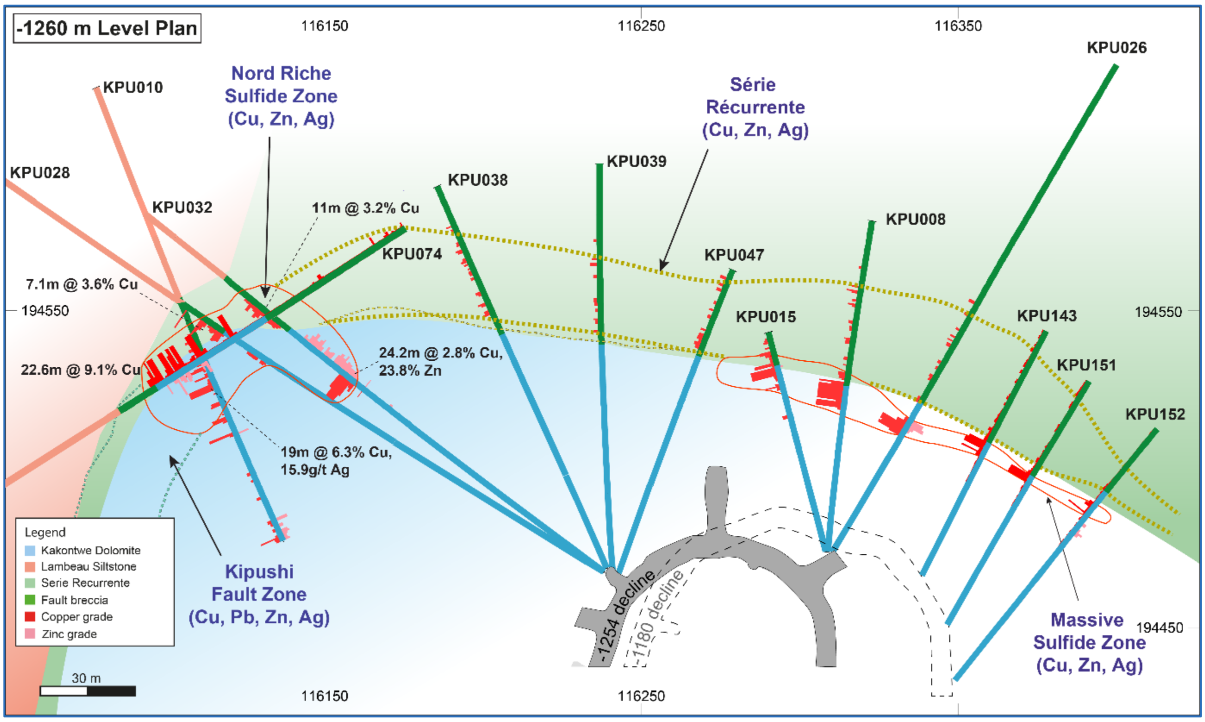
10.2.8 Copper Rich Zones of the Nord Riche, Série Récurrente, Fault Zone and the Splay

A plan projection of KICO drilling in the Copper Nord Riche and Série Récurrente is shown in Figure 10.5. Holes were drilled to test interpreted down-plunge extensions below the level of historical mining in the Copper Nord Riche area. These holes intersected zones of disseminated and massive sulphides (chalcopyrite and sphalerite), as shown in section in Figure 7.23.

The Série Récurrente contains a westerly-plunging lense of high-grade copper-rich massive sulphide, as described in Section 7.4.8, that extends from the Série Récurrente into the Upper Kakontwe. Drilling by Gécamines intersected this zone up-plunge, but it was not mined. KICO drilling in both the 2014 and 2017 campaigns intersected this massive mineralisation, with the extent shown in Figure 10.6. The mineralisation appears to pinch and swell, therefore potential extension to the east is still possible.

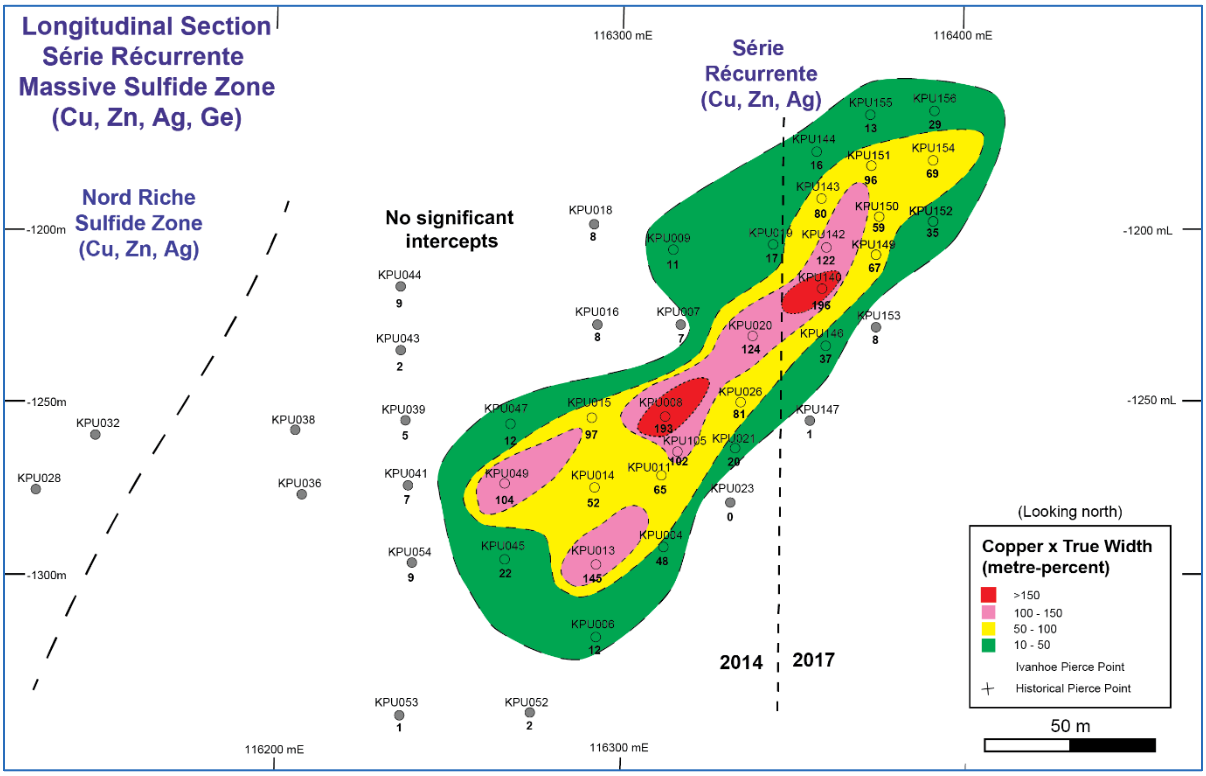
The disseminated style of 1–2% Cu mineralisation within the Série Récurrente can also be seen in Figure 10.5.

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



Source: Ivanhoe Mines (2018).

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

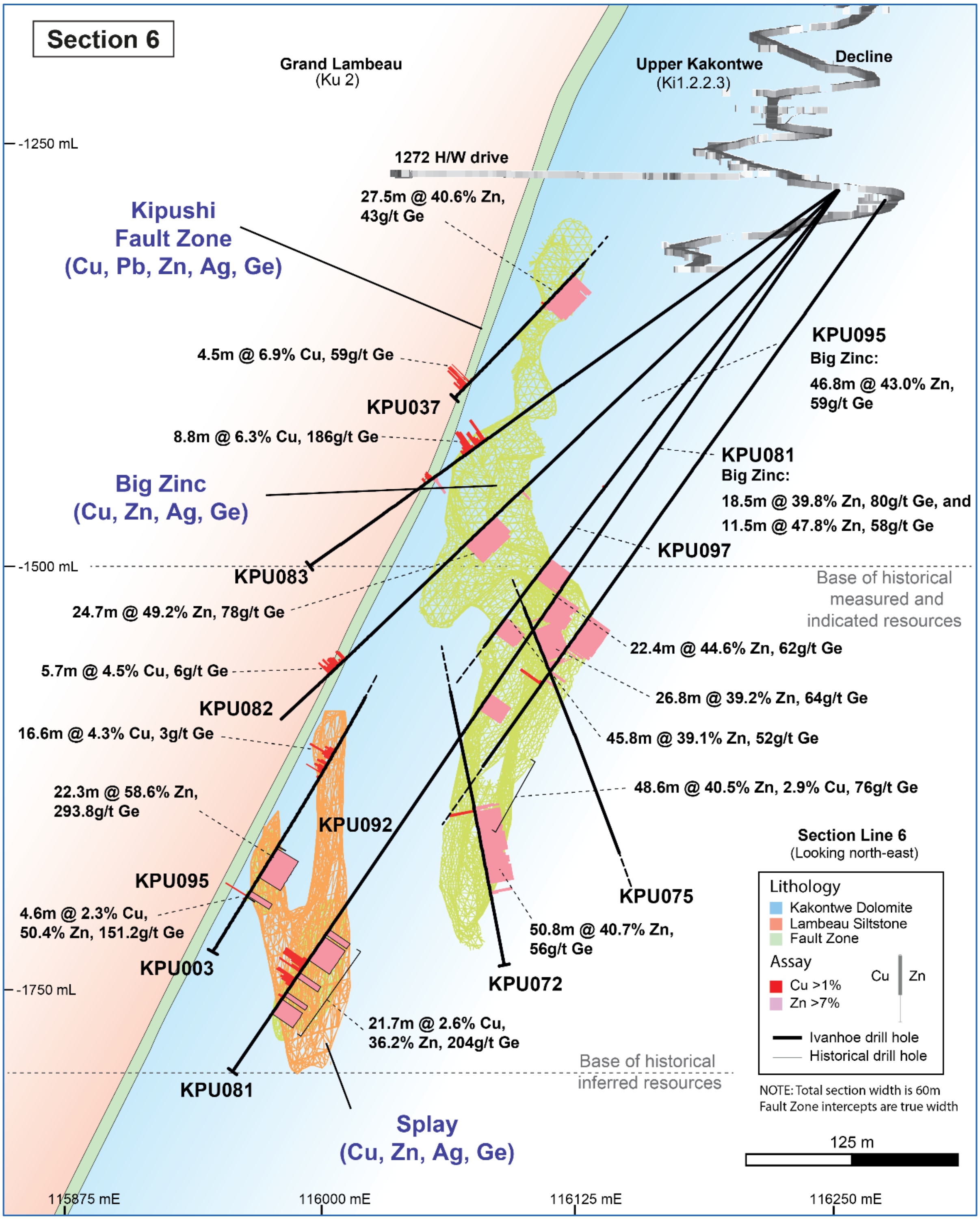


Source: Ivanhoe Mines (2018).

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

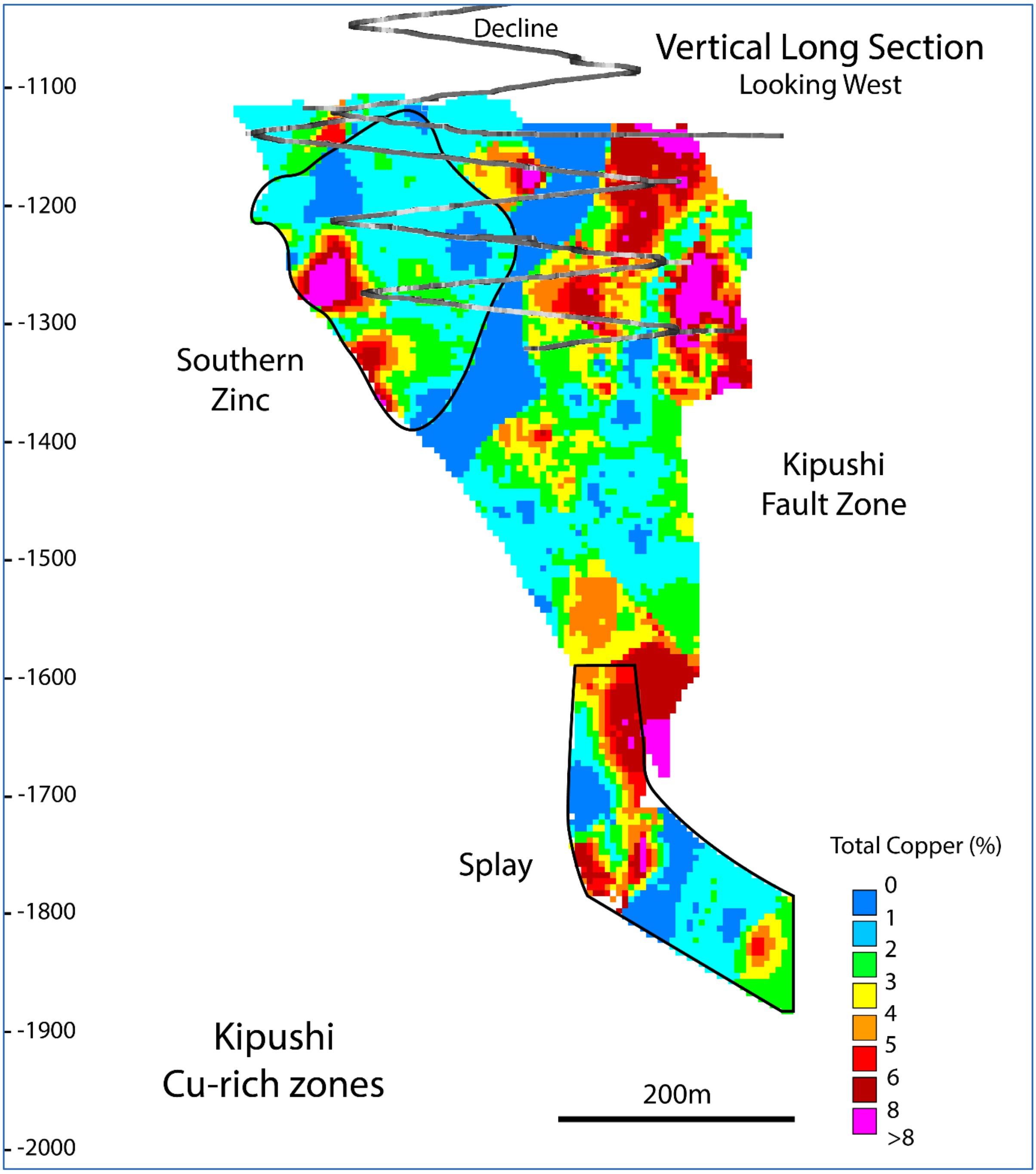
The copper dominant Fault Zone along with the associated copper mineralisation of the Southern Zinc and Splay zones are shown in the vertical long section in Figure 10.8. The southerly plunge to the mineralisation is clearly evident, with the two high-grade trends associated with the Big Zinc and Southern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

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



Source: Ivanhoe Mines (2018).

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



Source: Ivanhoe Mines (2018).

10.2.9 QP Comment

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

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

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Gécamines Sampling Approach

Sampling by Gécamines was selective and lower-grade portions of the mineralised 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 mineralisation ranging from less than 1 m to greater than 10 m in length with an average length of 3.44 m and divided into three categories (copper–copper/zinc, zinc, and copper–lead–zinc) and sampled.
- Waste material was not sampled.
- Remaining half core was placed in core trays and stored.
- Aggregated half core samples were sent to the Gécamines laboratory for crushing, splitting, milling, and sieving.

11.2 Gécamines Sample Preparation and Analytical Approach

All of the historical assays on samples generated by Gécamines drilling at Kipushi are believed to have been carried out at the Gécamines mine laboratory at Kipushi. Mr M Robertson from 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 was 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

Source: MSA, 2013.

11.3 KICO Sample Preparation Methods

All sample preparation, analyses and security measures were carried out under standard operating procedures set up by KICO for the Kipushi 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 mineralisation styles and lithological contacts. From hole KPU051 onwards, the nominal sample length was adjusted to 2 m for all zones with allowance for reduced sample lengths to honour mineralisation styles and lithological contacts; sample lengths within the copper-rich zones are typically 1 m or less. Following sample mark-up, the drill cores were cut longitudinally in half using a diamond saw. Half core samples were collected continuously through the identified mineralised zones.

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

Samples were dried at between 100°C and 105°C and crushed to a nominal 70% passing 2 mm, using either a TM Engineering manufactured Terminator jaw crusher or a Rocklabs Boyd jaw crusher. Subsamples (800 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 pulverised test, and 10 g of wet material was used for the wet pulverised test.

Subsamples collected for assaying and witness samples comprise the following:

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

Figure 11.2 Containerised Sample Preparation Facility at the Kolwezi Laboratory

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

Source: MSA, 2013.

Figure 11.3 Sample Preparation Facility at the Kamoa-Kakula Laboratory

A Drying oven

B Crusher and riffle splitter

C Crushers

D Labtech Essa LM2 pulverisers

E Dust filtration system

Source: MSA, 2013.

11.4 KICO Analytical Approach

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

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

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

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

Source: MSA, 2013.

11.5 Quality Assurance and Quality Control

11.5.1 QA/QC Approach

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

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

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

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

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

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

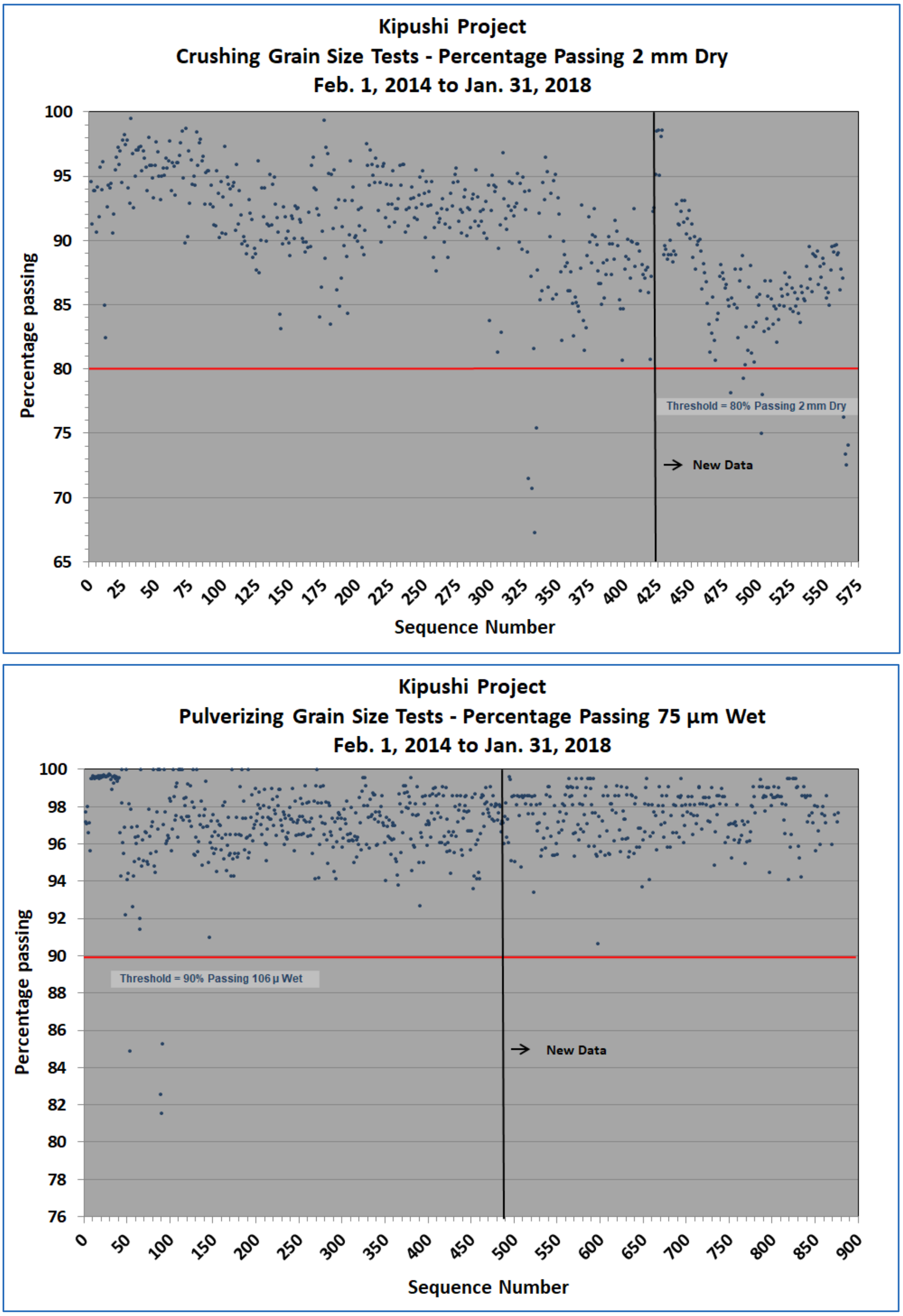
All of the sample batches submitted to BVM had approximately 5% CRMs, 5% blanks, and 5% crushed reject duplicates inserted into the sample stream.

11.5.2 Laboratory Performance

11.5.2.1 Sample Preparation

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

Figure 11.5 Crushing and Pulverising Grain Size Monitoring Charts



Source: Sketchley (2018).

11.5.2.2 Certified Reference Materials

CRMs were sourced from a number of independent commercial companies:

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

The AMIS, CCRMP, and OREAS series were used up to early 2015, and the KIP series preferentially thereafter. As the KIP series of CRMs was introduced late in the drilling programme, the results are of limited applicability for the entire data set. The CRMs were used to monitor the accuracy of laboratory assay results. Certified mean values and tolerance limits derived from a multi-laboratory round robin 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. The results of the CRM programme for the main elements of economic interest are shown in Table 11.3. Summary charts for zinc and copper are shown in Figure 11.7.

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

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

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

Table 11.1 Commercial CRMs Used in the KICO Drilling Programme

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

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

CRM	Zn (%)	Cu (%)	Pb (%)	Ag (AR) (ppm)	Ag (ppm)	Ge (ppm)	Au (FA) (ppb)	S (%)	As (ppm)	Cd (ppm)	Co (ppm)	Hg (ppm)	Re (ppm)
AMIS 83	–	–	–	–	–	–	–	–	–	–	–	–	–
AMIS 84	–	–	–	–	–	–	–	20.06	–	–	–	–	–
AMIS 144	–	–	–	–	–	–	–	–	–	–	–	–	–
AMIS 147	29.05	–	3.32	–	62.8	–	360	–	–	647	–	–	–
AMIS 149	–	–	–	–	–	–	–	–	–	–	–	–	–
AMIS 153	8.66	–	1.02	19.90	–	–	230	6.00	–	–	–	–	–
CZN4	55.07	–	–	–	51.4	–	–	33.07	–	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	–	–	–	–

Note: AR = Aqua Regia; FA = Fire Assay.

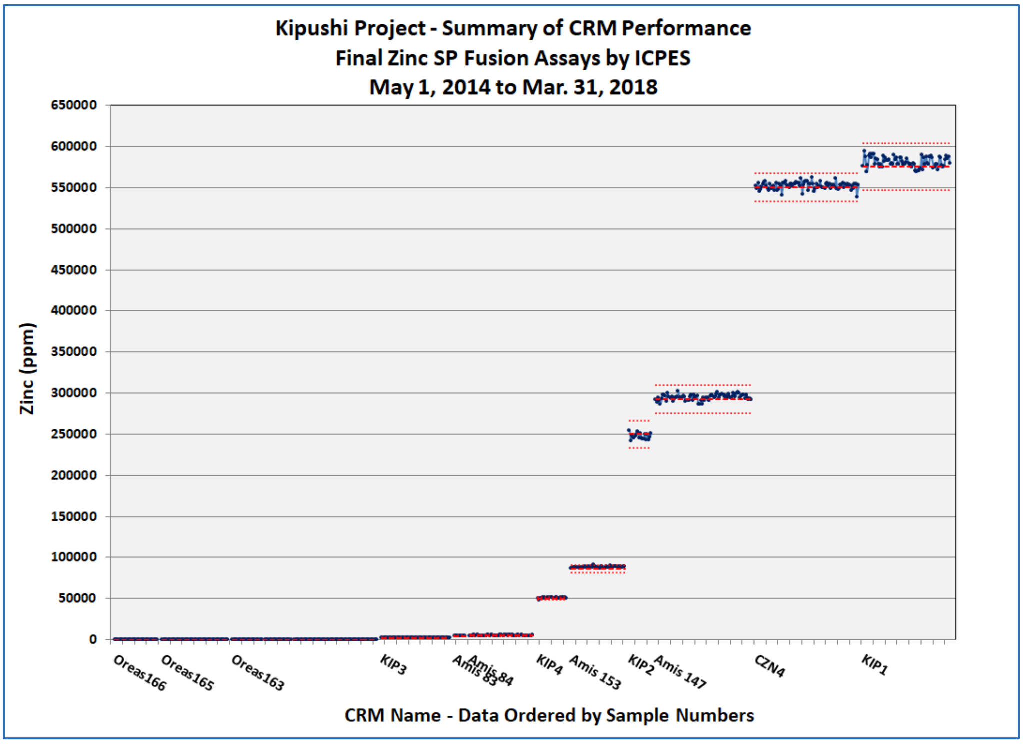
Table 11.3 CRM Performance for the Main Elements of Economic Interest

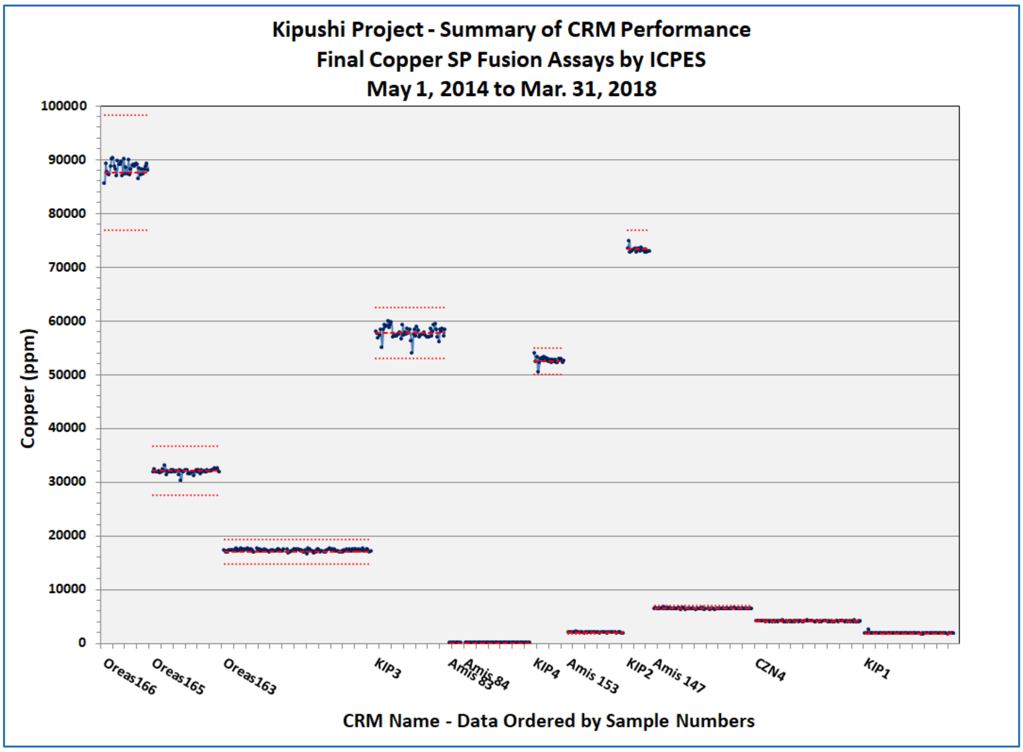
Element	Accuracy and Precision	Failures
Zn	Mean values within 2% of the certified values and RSD values <2%.	CZN4 and Amis 147 each had one positive failure. Re-assays addressed the CZN4 failure, whereas the one for AMIS 147 remains and is most likely due to a mix-up with a routine sample as the multi-element 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 strong negative bias due to the partial digest of the method. Due to the classification of Kip 2 as Provisional, there are no tolerance limits to classify samples as passed or failed
Ag (SFP)	Accuracy and precision for the AMIS and CZN CRMs is poor. AMIS 147 displays a negative bias of 6% and a RSD of 8%. CZN4 shows a negative bias of <2% and a RSD of 9%.	A number of negative failures remain for AMIS 147, with one likely due to a sample mix-up as the multi-element signature does not match any the CRMs. Re-assays returned values well below the range of the method for the surrounding routine samples; therefore the impact of the failures is regarded as negligible. CZN4 displays multiple negative failures due to poor resolution of the method.
Ge	Accuracy and precision for all 3 CRMs is poor.	KIP 1 displays no failures despite a strong negative bias of almost 11% in the first sampling campaign and no negative bais in the 2018 data. The single KIP 2 result is a marginal negative failure in phase 1 and has three failures, related to a positive bias and wider tolerance limits from the RR certification programme. KIP 4 displays one positive failure and poor precision, in each phase, due to the low value.
Au (FA)	Accuracy and precision for all CRMs tends to be poor.	AMIS 147 displays 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. Gold assays were discontinued during the 2018 drilling programme.

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

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

Figure 11.6 CRM Performance Summary for Zinc and Copper





Source: Sketchly (2018).

11.5.2.3 Blanks

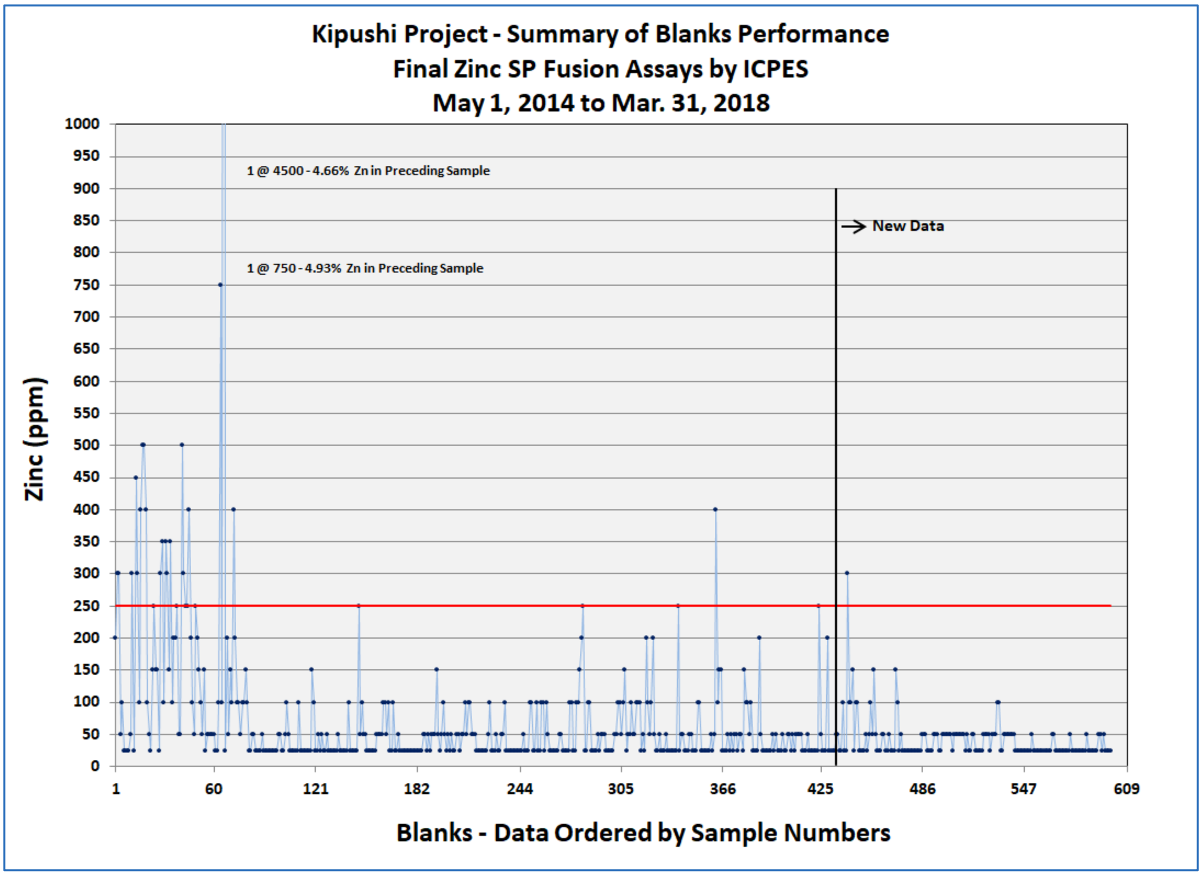
Locally obtained barren coarse quartz vein material was used to monitor contamination and sample mix-ups (Figure 11.2). This material was previously analysed in separate programmes (both Kipushi re-sampling and Kamoa-Kakula programmes) to ensure that it was barren of the elements of interest. Analytical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

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

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

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

Figure 11.7 Blank Sample Performance for Zinc by Sodium Peroxide Fusion



Source: Sketchley (2018).

11.5.2.4 Duplicates

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

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

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

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

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

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

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

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

11.5.2.5 Second Laboratory Check Assay Programme

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

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

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

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

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

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

11.5.3 Conclusions

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

- The results of the QA/QC programme demonstrate that the quality of the assay data for zinc, copper, and lead is acceptable for supporting the estimation of Mineral Resources. Higher-grade assays for silver, germanium, and gold are useable, but the limitations in the quality of the data should be taken into account.
- The second laboratory check assay programme conducted by Intertek validated the original BVM assays for most elements. Any future checking work should continue to use the Intertek laboratory; however, issues with carry-over need to be re-emphasised.
- Sample material for the second laboratory check assay programme was sourced from archived pulps (i.e. not the same pulps assayed by BVM) stored at the Kolwezi sample preparation facility. Future check assays should be conducted on the assay pulp residues remaining from the BVM assays.
- Gécamines did not carry out routine check assaying. Check assays were only carried out when visual grade estimates did not correspond with the laboratory results. Gécamines protocol for internal check sampling is unknown and there was no check assaying or sampling by an independent external laboratory.
- No data are available for QA/QC routines implemented for the Gécamines samples 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 mineralised core material was retained on site in a core storage building. The rejects and pulps were also stored, but over the years many were destroyed or lost.

KICO maintains a comprehensive chain of custody 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-Kakula Project facility. Core samples are delivered from Kipushi to the sample preparation facility by company vehicle. On arrival at the sample preparation facility, samples are checked, and the sample dispatch forms signed. Prepared samples are shipped to the analytical laboratory in sealed sacks that are accompanied by appropriate paperwork, including the original sample preparation request numbers and chain-of-custody forms.

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

12 DATA VERIFICATION

A comprehensive re-sampling programme was undertaken on historical Gécamines drillhole core from the Big Zinc 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 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. 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

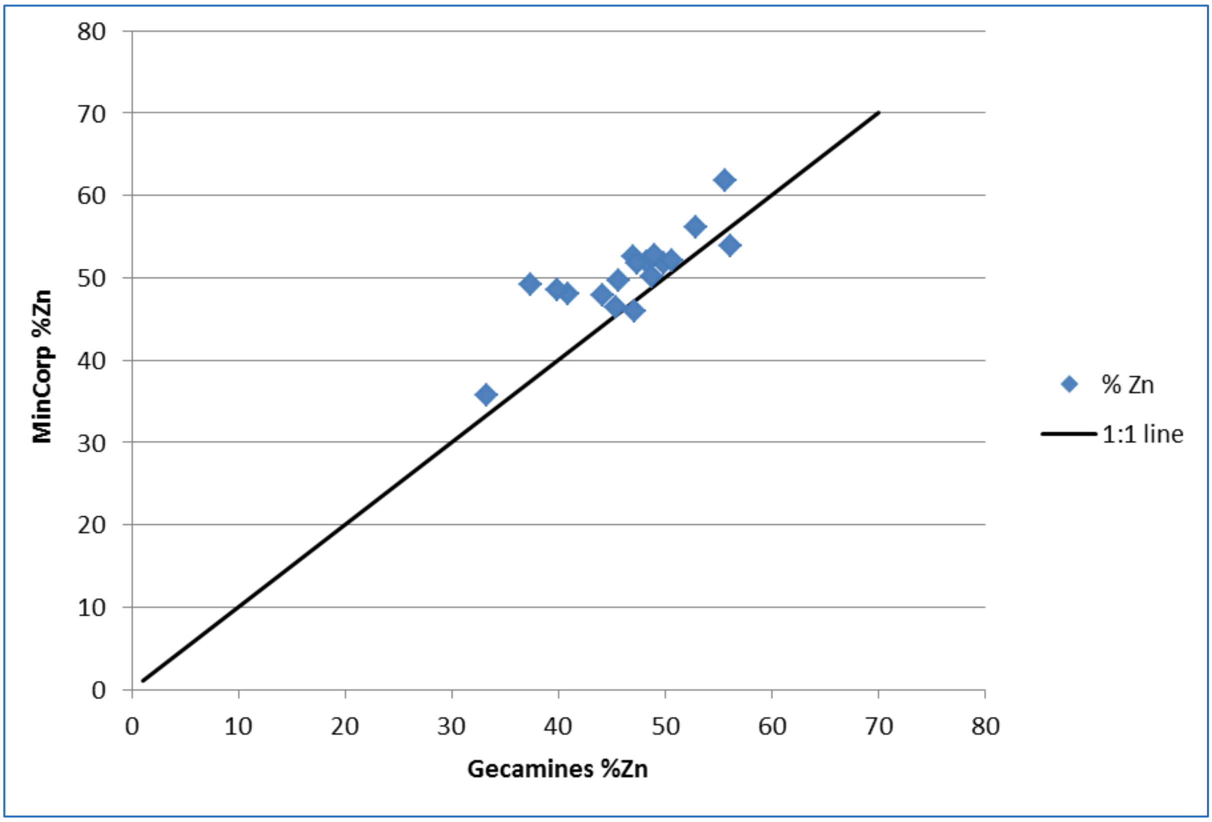


Figure by MSA, 2014.

12.2 Big Zinc and Fault Zone Re-sampling Programme

12.2.1 Sample Selection

An initial site visit to Kipushi was undertaken from 20 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 had composite assay data results and individual sample assays were not available or were not captured.

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

Re-sampling of the drill core was supervised by the 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, although not true thickness.
1270	5	-55; -65; -75	-30	22	Intersects upper part of Big Zinc, 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.
1270	9	-40; -75	-40	25	Medium Cu zone on Fault Zone; intersects entire middle zone of Big Zinc; (-85 hole core not available therefore not an option).
1270	11	-45; -65	-25	15	Intersects upper part of Big Zinc; includes narrow zones of high Cu.
1270	13	-65	-75	19	Narrow zones of high Cu; intersects lower part of Big Zinc.
1270	15	-20	-40	12	High Cu in Fault Zone; intersects middle zone of Big Zinc.
1270	17	-70	-75*	16	Intersects lower part of Big Zinc.

* 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 QA/QC

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

CRM over-reporting failures for Zn and S were observed in the initial BVM assays, which led to a re-assay of Zn and S for all 457 samples. The over-reporting was confirmed by the results of 128 pulp splits analysed at a second laboratory (Intertek Genalysis in Perth). Although an improvement in the accuracy of results was noted in the re-assays, CRM failures for Zn and S were still observed and this was brought to the attention of BVM who re-analysed 120 samples for Zn and S using a modified approach. These results were regarded by the QP as acceptable. BVM was then requested to re-analyse all 457 samples for Zn and S in order to provide a "clean" set of data. These final re-assays, together with the other multi-element results, which were accepted from the initial BVM work, comprise the final assay dataset for the re-sampling programme. A comparison of mineralised intersections, at a cut-off of 7% Zn, between historical and re-sampling results is shown in Table 12.3. The comparison revealed 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 mineralisation.
- 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.

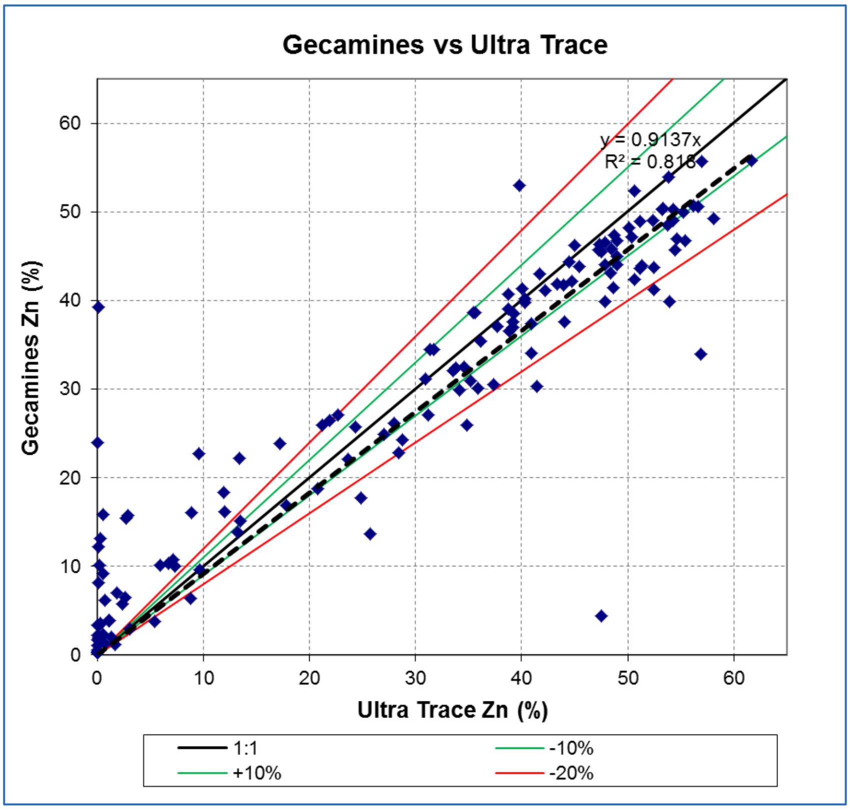
Table 12.3 Comparison of Mineralised Intersections between Gécamines and the re-Sampling Programme Using a Cut-off of 7% Zn

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



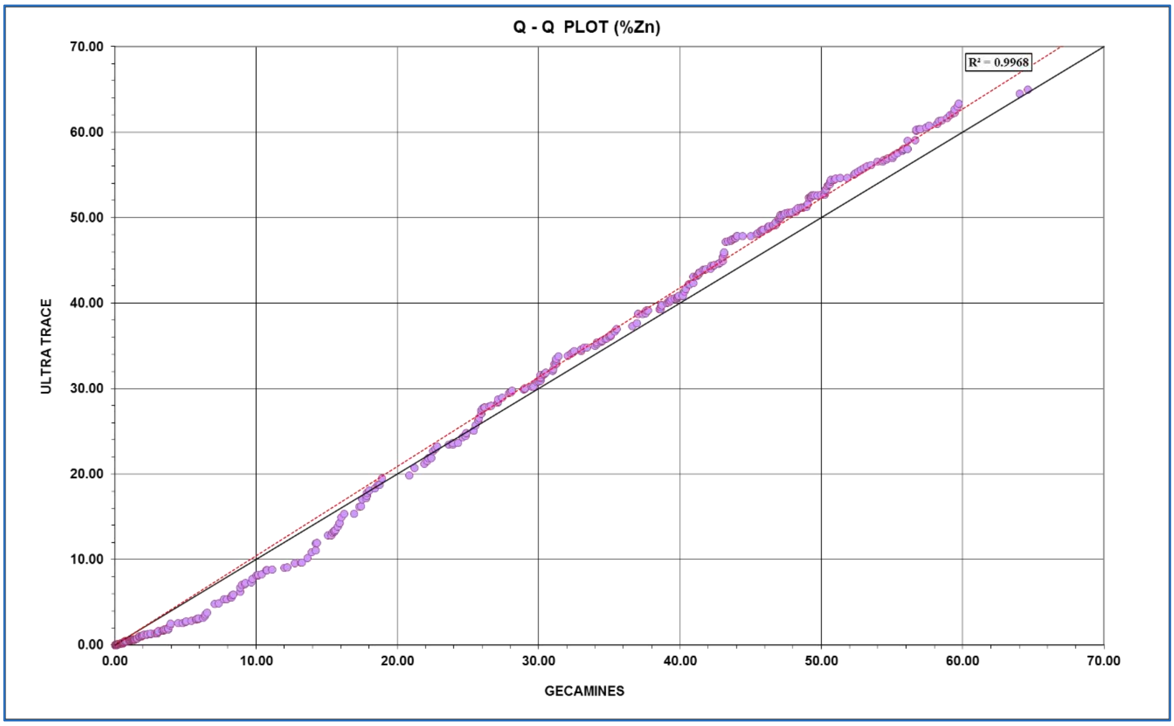


Figure by MSA, 2014.

Figure 12.3 Histogram Plot of BVM Ge Results

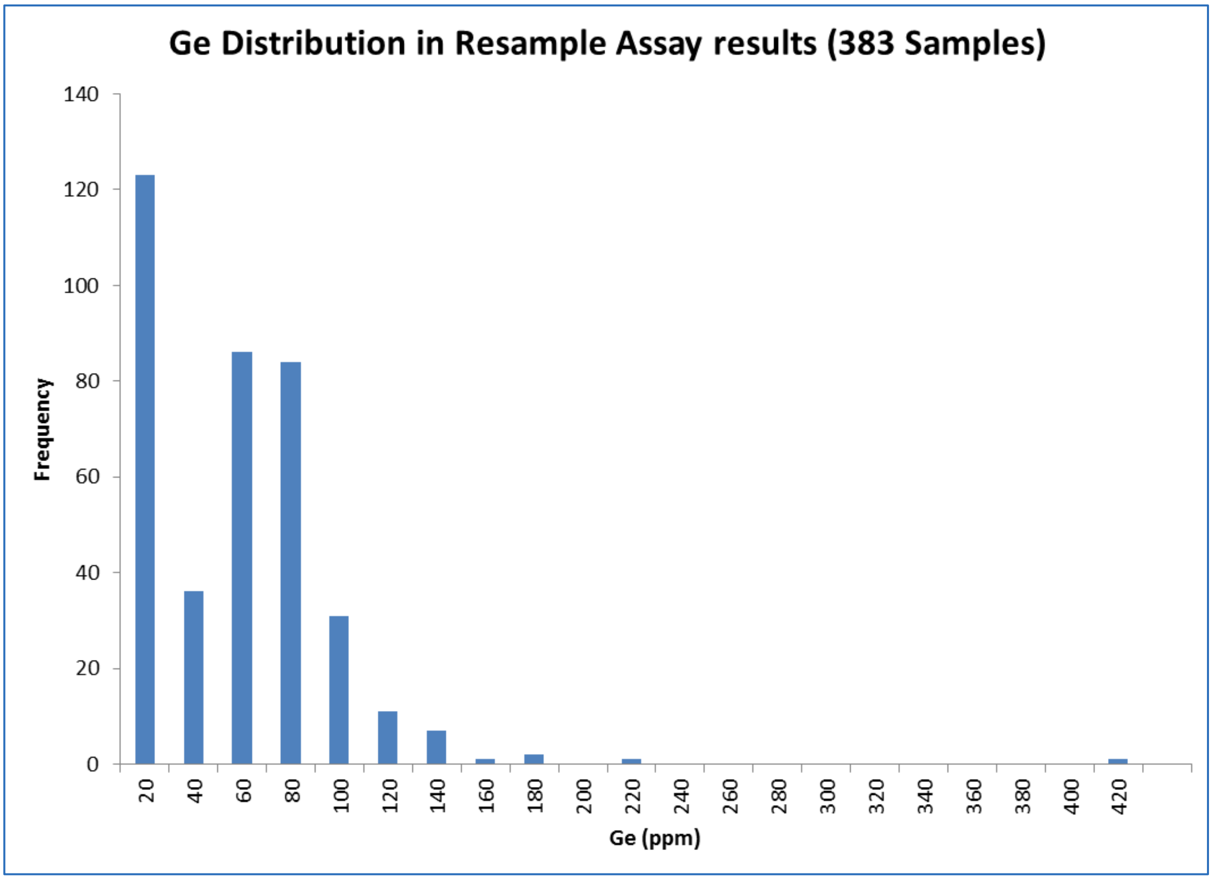


Figure by MSA, 2014.

12.2.4 Density Considerations

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

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

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

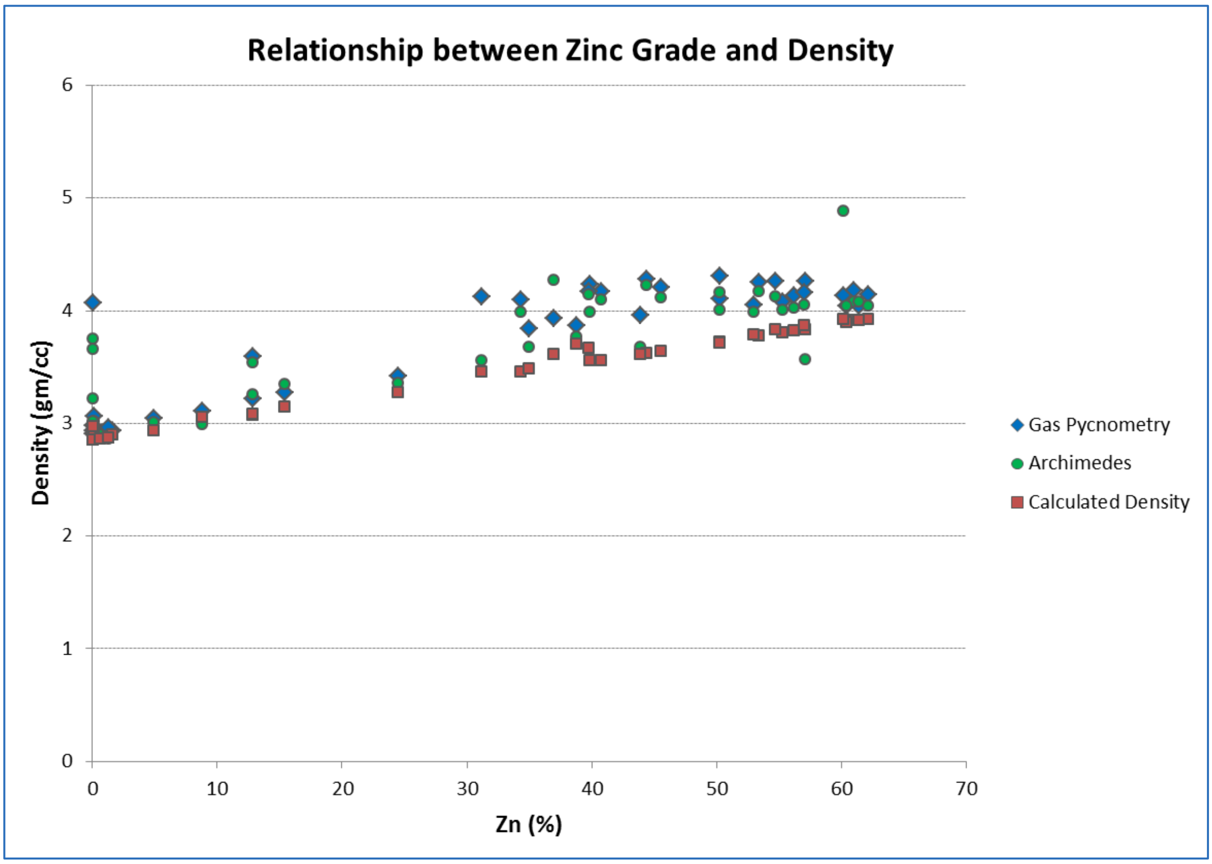


Figure by MSA, 2014.

12.3 Re-logging Programme

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

12.4 Twin Hole Drilling Programme

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

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

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

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

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

Table 12.4 Kipushi Twinned Holes

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

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

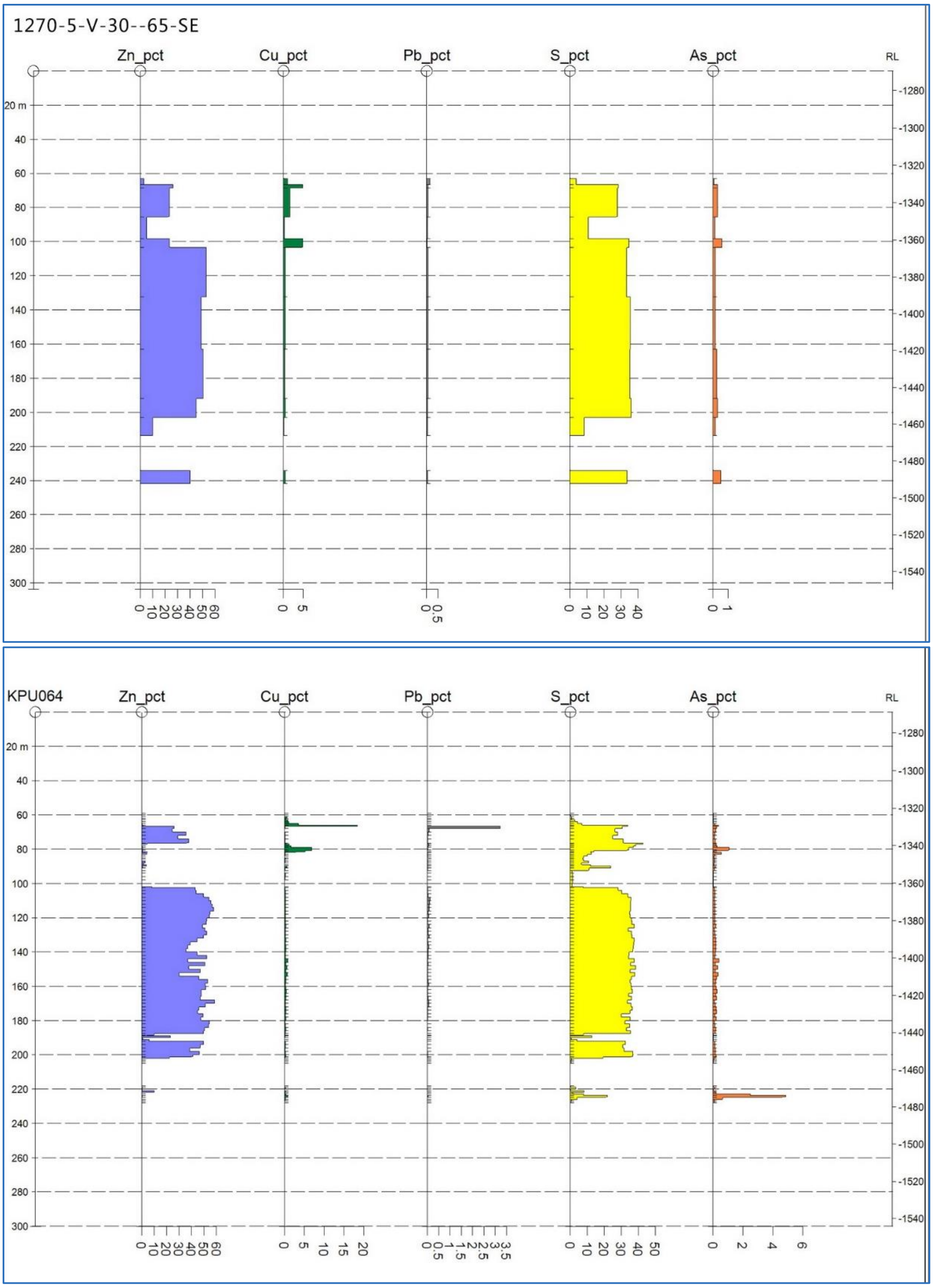


Figure by MSA, 2014.

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

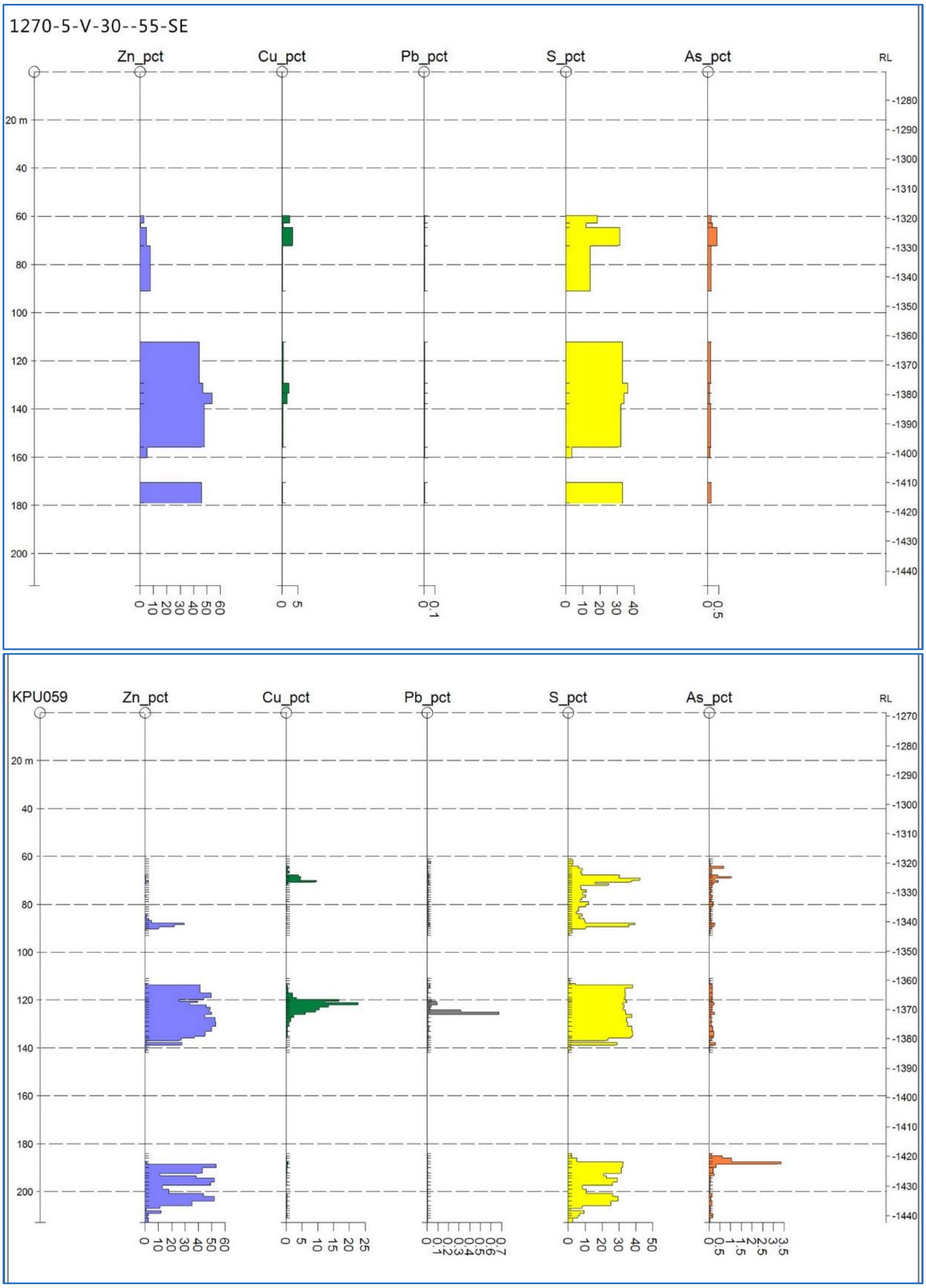


Figure by MSA, 2014.

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

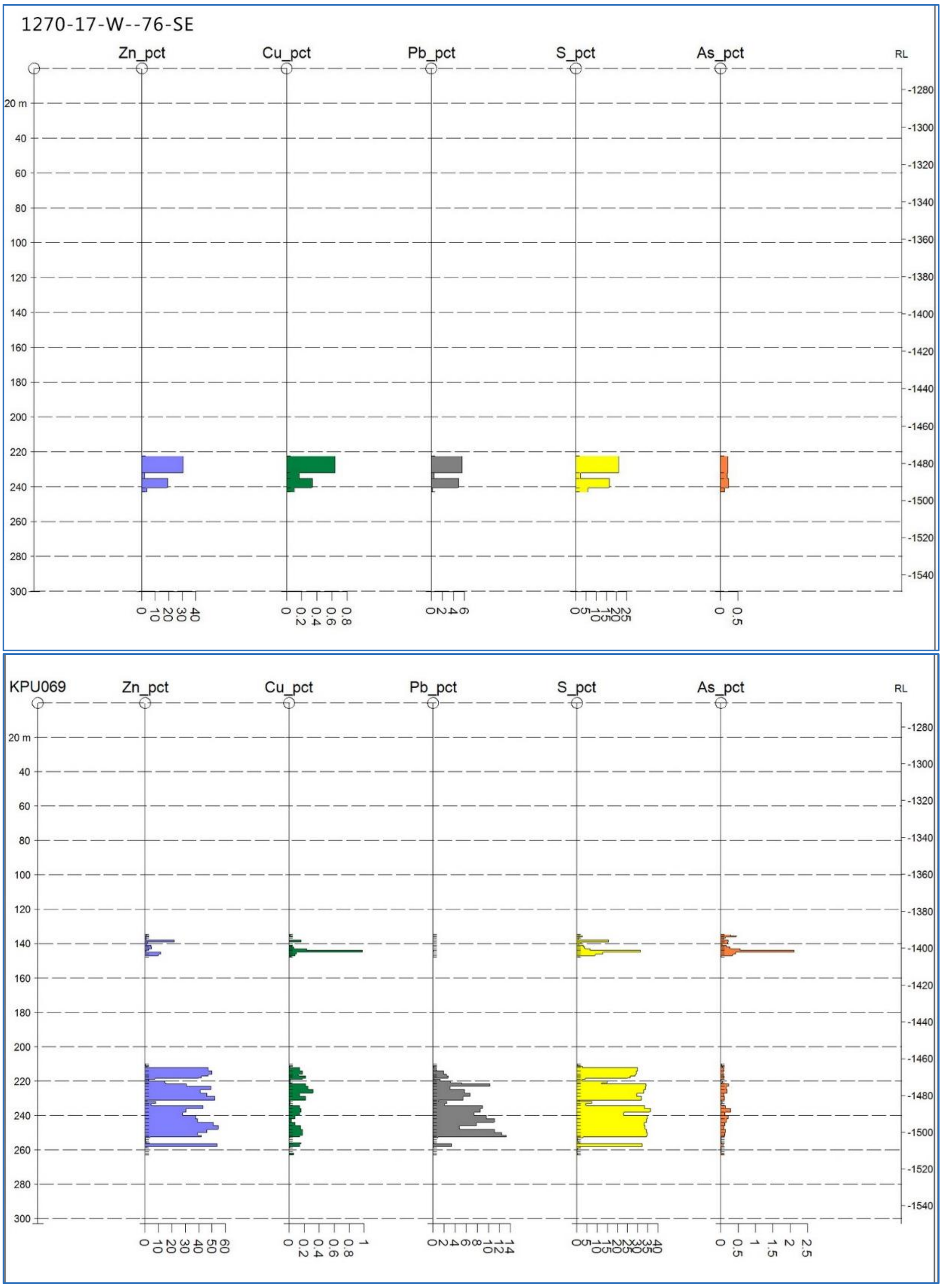


Figure by MSA, 2014.

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

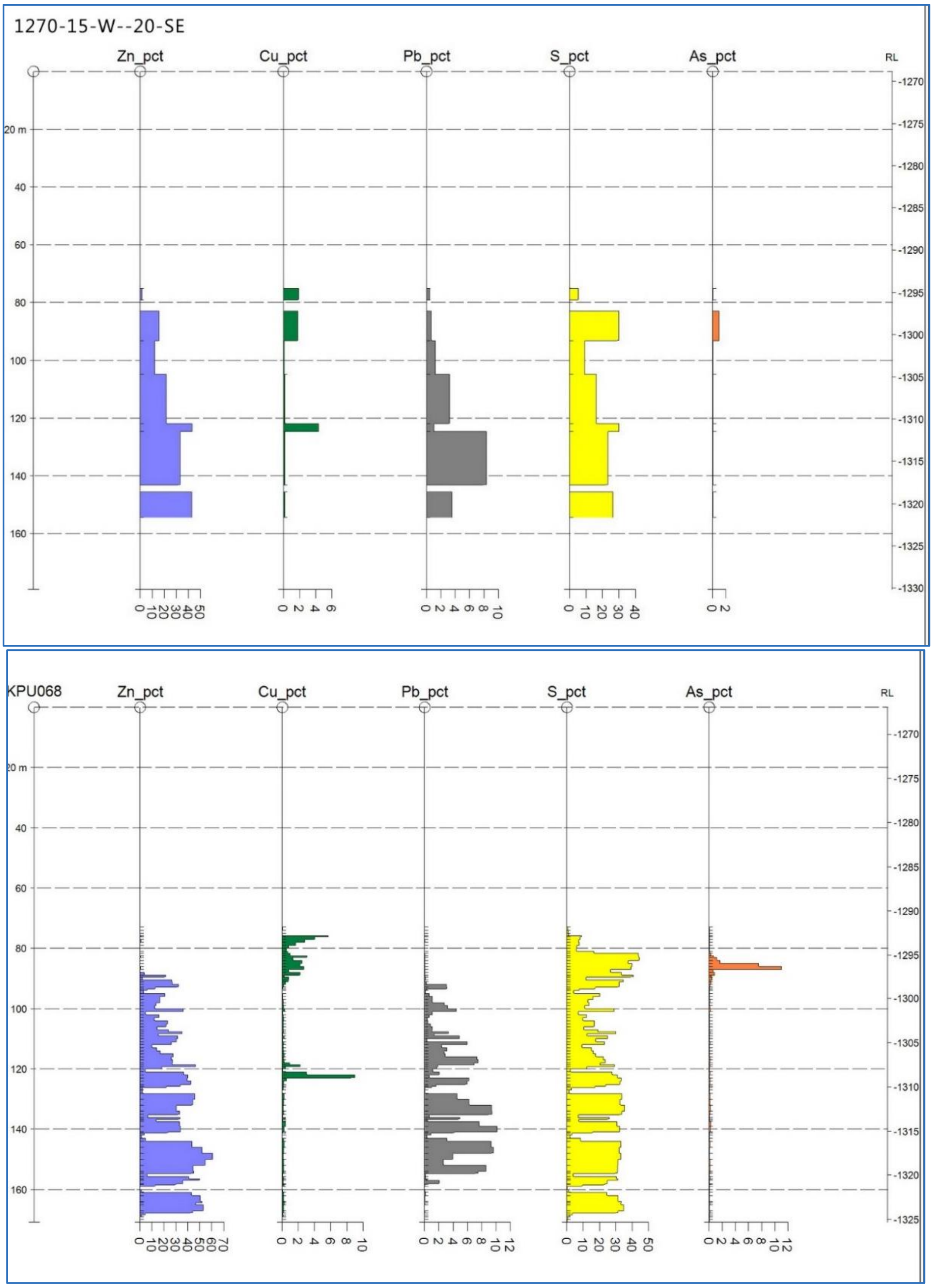
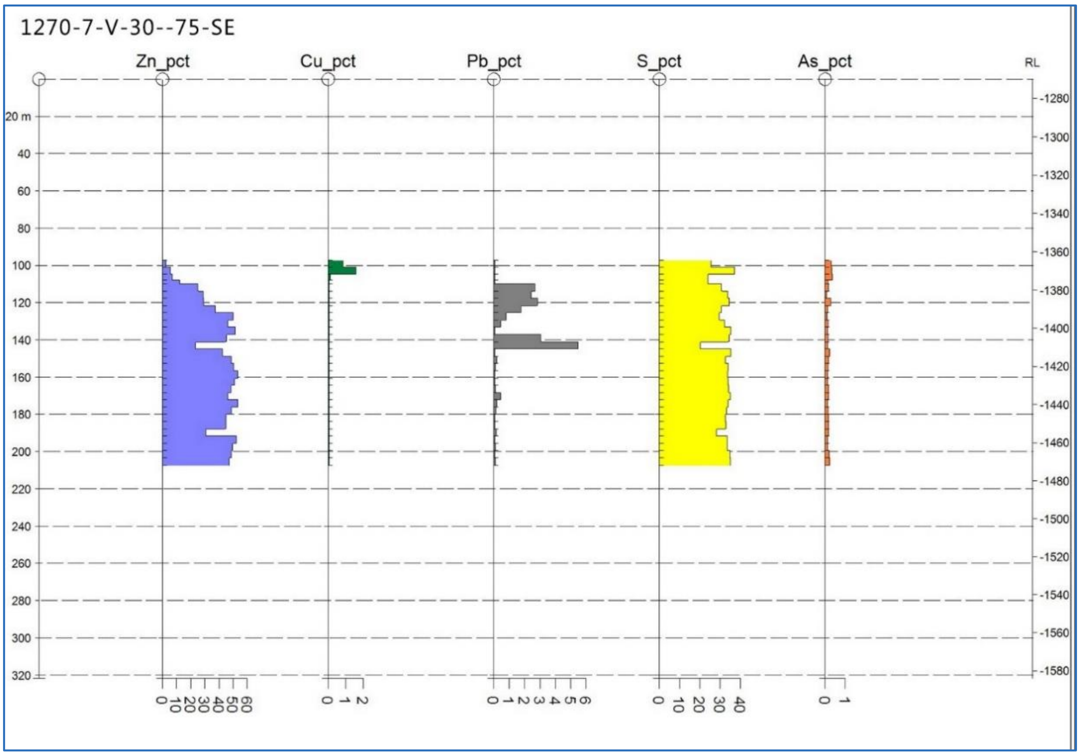


Figure by MSA, 2014.

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



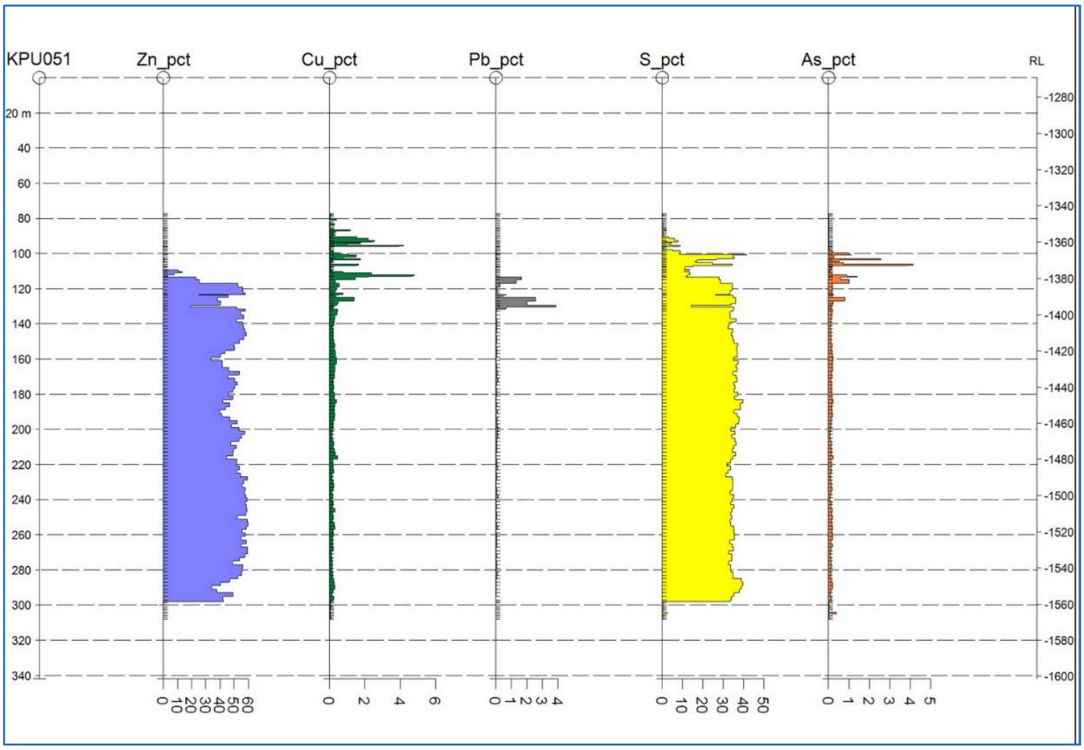
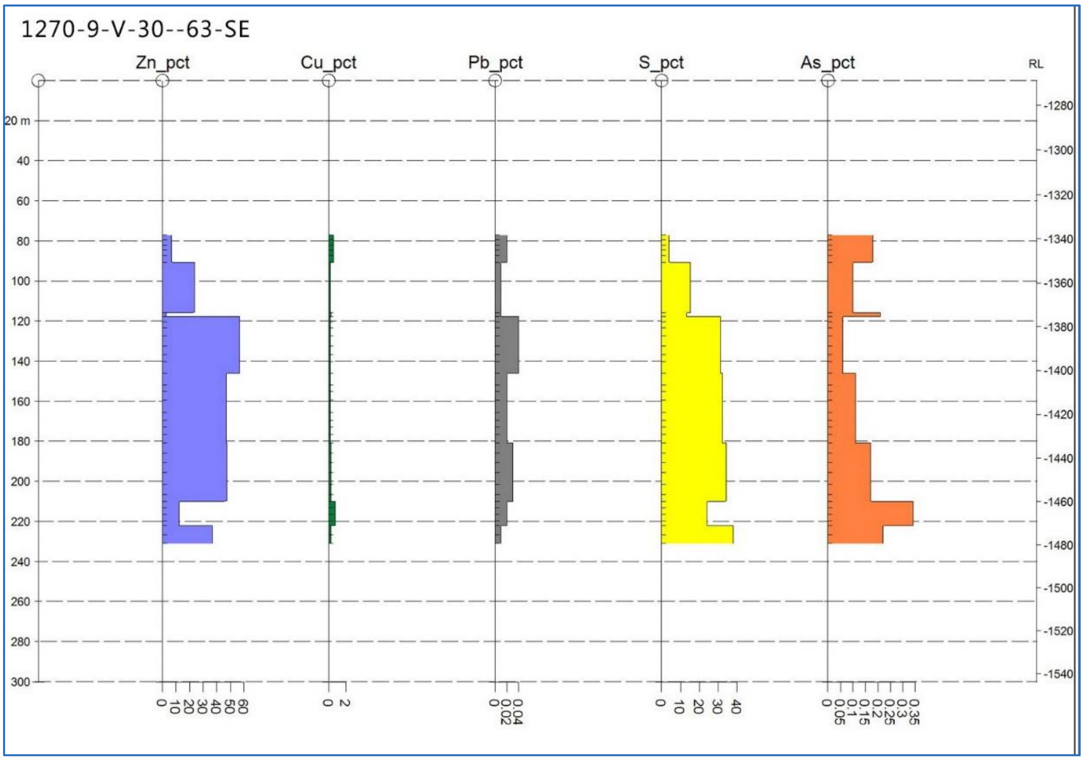


Figure by MSA, 2014.

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



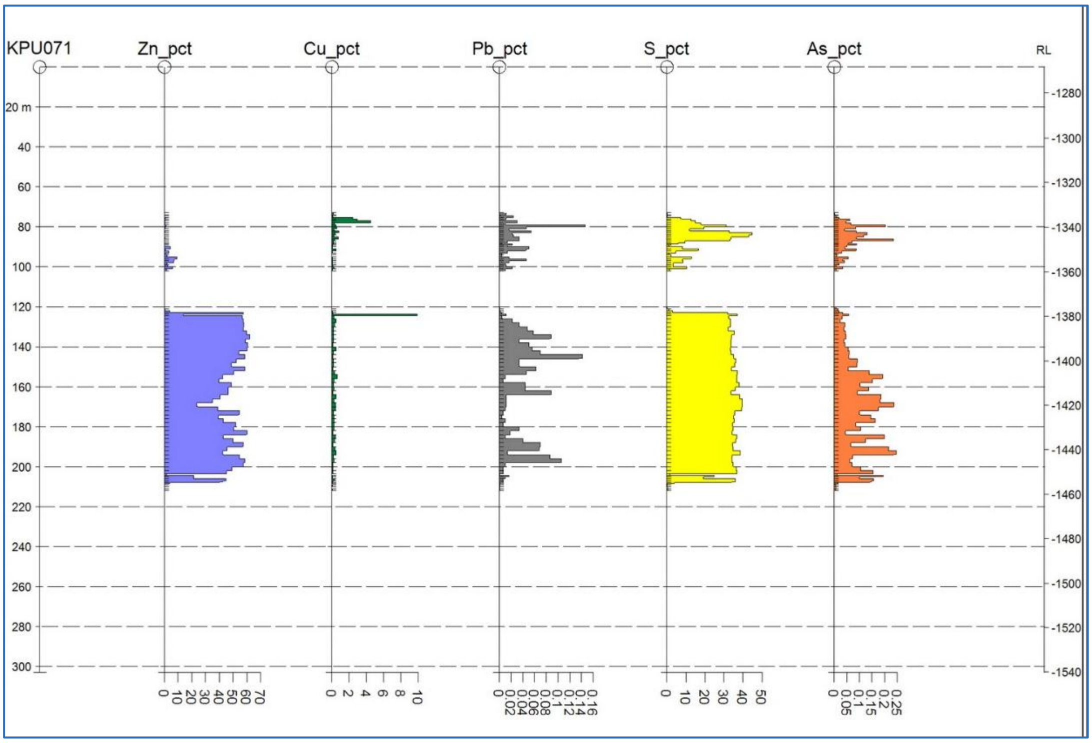


Figure by MSA, 2014.

12.5 Visual Verification

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

12.6 Data Verification Conclusions

In the opinion of the 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 QA/QC control.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

This section has not been changed from the Kipushi 2017 PFS and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Kipushi 2017 PFS.

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

Metallurgical testwork programmes were completed on drill core samples of known Kipushi mineralisation between 2013 and 2017, for the various project redevelopment study phases. These investigations were focused on metallurgical characterisation, and flowsheet development, for the processing of material from the Big Zinc. 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 then undertaken by Ivanhoe. The review suggested that given the favourable density differences between massive sulphides and gangue material, Heavy Media or DMS was considered a highly likely alternate to flotation, potentially providing lower capital and operating costs, and this formed the basis for the Kipushi 2016 PEA.

A Metallurgical sampling and testwork campaign with additional tests were conducted between 2016 and 2017, on drill cores intercepts constituted for variability composite samples and a development composite sample. Gravity separation circuit (DMS and Spiral Plant) results indicated that the DMS was effective in rejecting dolomite material however the concentrate collected all heavy minerals, and higher base metal sulphide content in the feed, automatically reported to the concentrate, reducing the concentrate zinc grade.

A more robust PFS processing flowsheet was considered, that added a milling and flotation circuit at the back end of the Dense Media Separation (DMS) plant, to ensure that a consistently high-grade concentrate product could be produced.

13.1 Kipushi 2016 PEA Testwork

The information as presented herein, is an extension to the Kipushi 2016 PEA issued in May of 2016, the key difference being the change in processing method, from a solely gravity-based processing circuit (DMS and spirals) to one that uses a combination of; gravity (DMS) to partially beneficiate the ore (removal of dolomite) and a physiochemical process to remove copper, lead and pyrite from sphalerite (milling and flotation).

In 2013, approximately 60 kg of Kipushi quarter-core was delivered to Mintek (South Africa), for metallurgical testwork, including; mineralogy, comminution, gravity and flotation testing. The composite sample head analysis was: zinc (38%); lead (0.78%); copper (0.4%); sulphur (34%) and iron (12%). Mineralogy of the sample showed sphalerite being predominate (65.9%), followed by pyrite (24%), with galena and chalcopyrite present in minor quantities. The major gangue component was silica and carbonaceous minerals. The sphalerite and galena were coarse grained, grains up to 1 mm and 0.5 mm respectively, whilst chalcopyrite showed relatively fine grains, less than 0.04 mm in size.

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

Preliminary flotation tests indicated a zinc rougher recovery of 87%, at 56% Zn concentrate grade, at a P_{50} of 75 µm.

A second metallurgical sampling and testwork campaign was conducted in 2015, as part of the Kipushi resource development phase; with the Big Zinc being the primary focus of this programme. Six drillholes intercepting the Big Zinc were selected and intervals composited for metallurgical and mineralogical investigations. The samples came from hole numbers; KPU001, KPU003, KPU042, KPU051, KPU058, and KPU066. The location of these drill cores with reference to the Big Zinc are illustrated in Figure 13.3 below. The drill core for the composite was selected to represent all mineralisation types in the Big Zinc 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, with sections of core of known grade, selected accordingly.

Mineralogical investigations conducted on the composite head sample, identified the following economic minerals (in order of abundance): sphalerite (67%), galena (2%), and chalcopyrite (1%). The main gangue minerals in the sample were: dolomite (18%); followed by pyrite (8%) and quartz (3%). The head assay of the composite is presented in Table 13.1.

Table 13.1 Kipushi Composite Head Sample

	Zn %	Pb %	Fe %	Ca %	Si %	Cu %	Mg %	S %
Avg. Head Assay	40.10	1.45	5.97	6.20	1.73	0.27	3.55	25.45

In addition to the earlier flotation test, the testwork programme was expanded to determine whether gravity processing alone, was a viable upgrade step. To this end, 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.

HLS or sink-float analysis is a laboratory scale characterisation method, which uses heavy organic liquids, mixed to different densities, to determine sample gravity separation across a range of densities. Practical mine operation would generate a particular density of media using FeSi (Ferrosilicon powder) and perform the separation with the ore using cyclones.

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 using increments of 0.1 g/ cm^3.

Analysis determined that a density cut point of 3.1 g/cm^3 was optimal in all cases. The test results for these three test samples at this density are presented in Table 13.2.

Table 13.2 Circuit Performance Summary

Concentrate			
Size Fraction	**Mass Yield (%)**	**Zn Grade (%)**	**Zn Rec (%)**
-20+1 mm	70.1	55.5	95.5
-12+1 mm	68.7	53.3	90.0
-6+1 mm	65.1	55.7	89.2

Performance across the HLS and the shaking table was the same for all three crush sizes. The HLS circuit achieved an excellent 99% recovery, at a concentrate grade of approximately 55% Zn.

However, the shaking table testwork was not so good. The shaking table achieved 58% recovery, at a concentrate grade of approximately 56% m/m (d.b) zinc. The fact that the coarser crush size (-20+1 mm) produced less -1 mm fines, meant that this size had superior overall recoveries. The -20 mm sample had 10% of feed bypassing the HLS, compared to 22% and 32%, of the –12 mm and –6 mm samples respectively. The –20 mm crush size achieved an overall recovery of 95.5%, at a saleable concentrate grade of 55.5% Zn.

13.2 PFS Testwork

13.2.1 Sample Selection and Composition

In 2016, approximately 900 kg of half core from eight drillholes intercepting the Big Zinc were selected, and intervals composited, for a variability and flowsheet development testwork program ahead of the PFS. Core intercepts across the Big Zinc were sampled and constituted at various feed composition for variability tests. A 73 kg PFS development composite was also constituted for flowsheet development and the optimisation testwork program. The PFS development composite samples came from hole numbers; KPU001, KPU042, KPU085, and KPU086. The location of these drill cores with reference to the Big Zinc are illustrated in Figure 13.1 and Figure 13.2, whilst the associated core data is presented in Table 13.3.

The drill cores for the PFS composite were selected to represent all styles of mineralisation in the Big Zinc including, but not limited to: Massive Brown Sphalerite (MSB); Massive Sulphide Mixed (MSM); and Dolomite (SDO). Based on the assayed intervals of the resource drill core, core section from the various holes were composited to give a metallurgical testwork sample, grading around 32% Zn.

After selection, the cores were firstly packaged into plastic bags and then polywave bags. The polywave bags were packed into carton boxes, which were then foam filled to avoid abrasion shock between samples. The carton boxes were then inserted into metal trunks to ensure sample integrity and security in transit (air and road), between Kipushi and Mintek's laboratories in South Africa. Cores were drilled in 2014/2015 and received at Mintek in August 2016. At Mintek, the material was crushed to a product size of 100% passing 20 mm, for gravity separation tests, subsampled and further crushed to 100% passing 1.7 mm, and split into 1 kg flotation tests charges, and then stored in bags in freezers.

Figure 13.1 Metallurgy Sample Map

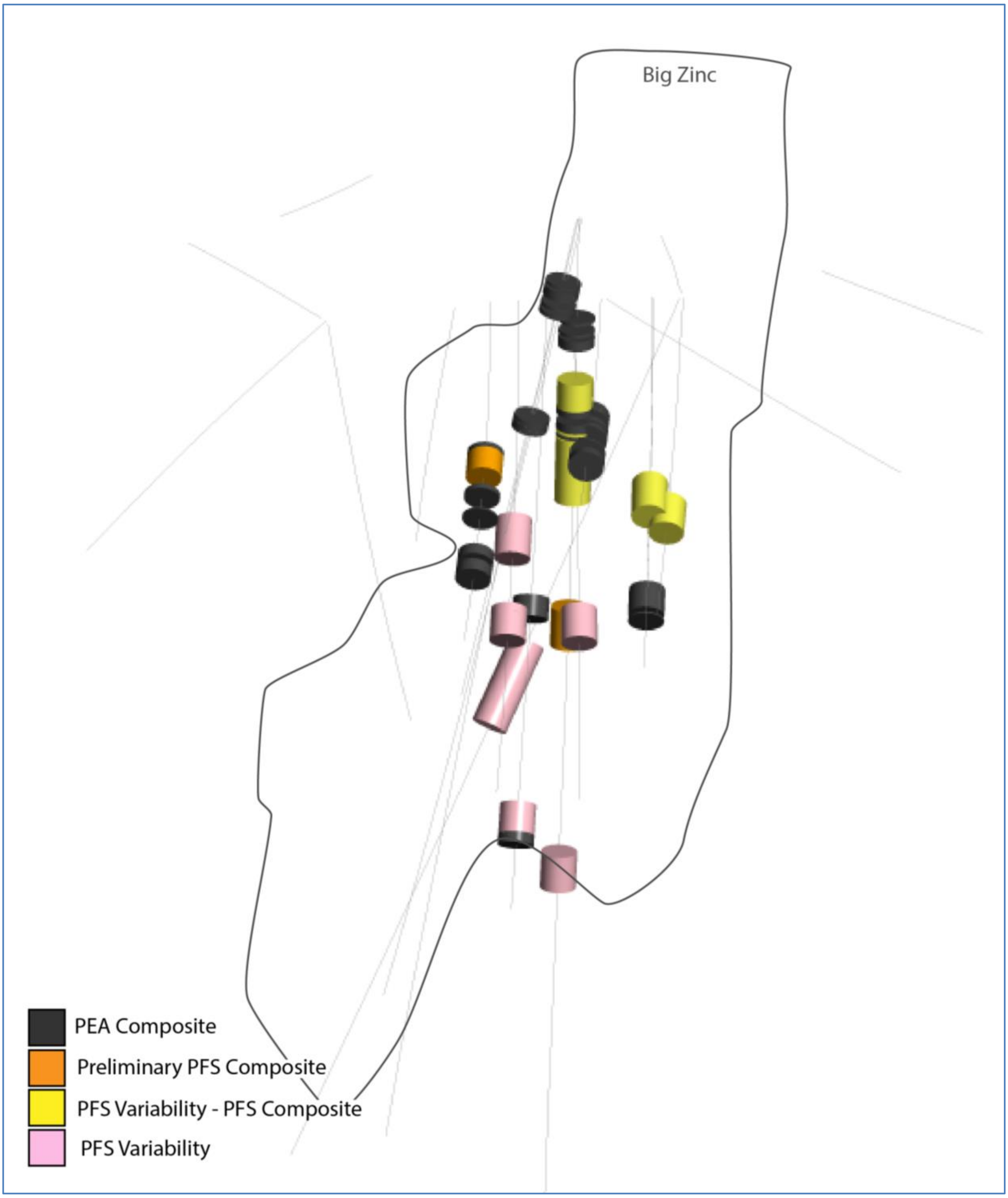


Image provided by Ivanhoe, 2018.

Figure 13.2 Kipushi 2017 PFS Metallurgy Sample Map

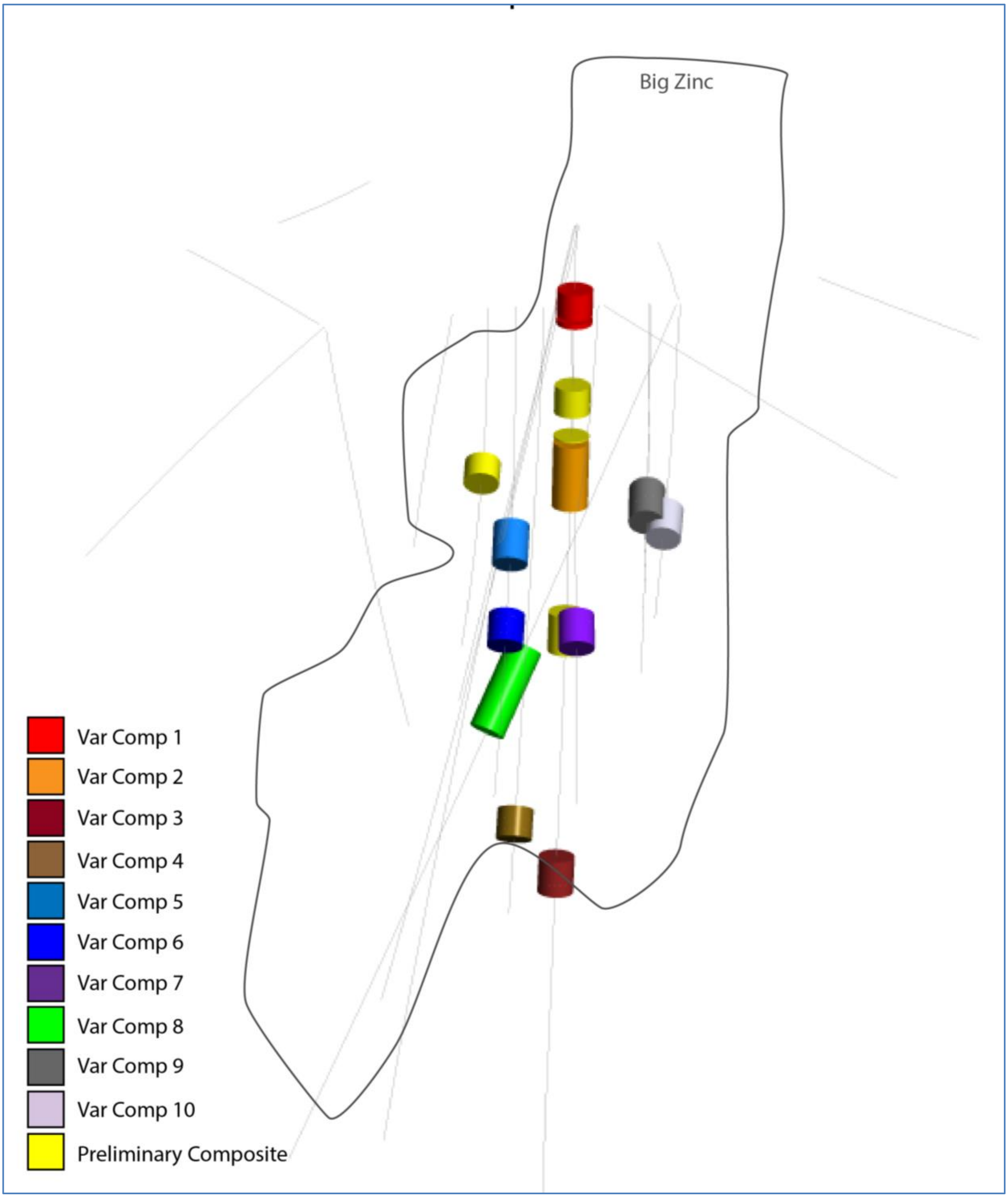


Image provided by Ivanhoe, 2018.

Table 13.3 Kipushi 2017 PFS Core Sample

<table>
<tr><th>Sample</th><th>Hole ID.</th><th>From (m)</th><th>To (m)</th><th>Interval (m)</th><th>Measured Weight (kg)</th><th>Sampled Weight (kg)</th><th>Mass Distribution (%)</th><th>Zn (%)</th><th>Pb (%)</th><th>Fe (%)</th><th>Cu (%)</th><th>S (%)</th></tr>
<tr><td>Var Comp 2</td><td>KPU001</td><td>130</td><td>168</td><td>25.5</td><td>80.9</td><td>40</td><td>54.79</td><td>32.98</td><td>0.05</td><td>6.09</td><td>0.38</td><td>21.98</td></tr>
<tr><td>Var Comp 9</td><td>KPU085</td><td>135.3</td><td>160</td><td>24.7</td><td>84.8</td><td>8</td><td>10.96</td><td>40.55</td><td>4.55</td><td>6.44</td><td>1.15</td><td>26.75</td></tr>
<tr><td>Var Comp 10</td><td>KPU086</td><td>144</td><td>165</td><td>21</td><td>68.1</td><td>3</td><td>4.11</td><td>35.25</td><td>6.69</td><td>6.77</td><td>0.22</td><td>25.05</td></tr>
<tr><td rowspan="3">Preliminary Composite</td><td>KPU001</td><td>96</td><td>127</td><td>23</td><td>81.7</td><td rowspan="3">22</td><td rowspan="3">30.14</td><td rowspan="3">38.85</td><td rowspan="3">0.05</td><td rowspan="3">11.58</td><td rowspan="3">0.37</td><td rowspan="3">31.75</td></tr>
<tr><td>KPU001</td><td>231</td><td>254</td><td>23</td><td>71</td></tr>
<tr><td>KPU042</td><td>105.7</td><td>120</td><td>14.4</td><td>60.5</td></tr>
<tr><td colspan="6">Head grade (calc.)</td><td>73</td><td>100</td><td>35.67</td><td>0.82</td><td>7.81</td><td>0.46</td><td>25.57</td></tr>
<tr><td colspan="6">Head grade (meas.)</td><td></td><td></td><td>32.73</td><td>0.72</td><td>6.79</td><td>0.42</td><td>24.53</td></tr>
</table>

The 73 kg PFS development composite was used for Kipushi 2017 PFS circuit development and optimisation tests, including mineralogy and flotation tests.

13.2.2 Head Analysis

All the eleven composites were crushed to -20 mm, subsampled and prepared for feed chemical analysis. Head assays are presented in Table 13.4.

Table 13.4 Kipushi 2017 PFS Core Head Assays

Sample	Zn (%)	Pb (%)	Fe (%)	Si (%)	Ca (%)	Mg (%)	Cu (%)	S (%)
Var Comp 1	30.08	1.64	17.93	4.96	1.59	0.83	0.41	33.25
Var Comp 2	32.98	0.05	6.09	0.26	8.38	4.89	0.38	21.98
Var Comp 3	35.90	0.07	12.80	0.23	3.97	2.75	0.87	28.95
Var Comp 4	44.30	0.05	13.10	0.05	1.84	1.30	0.23	34.80
Var Comp 5	23.70	0.05	12.70	5.21	6.57	3.74	0.21	24.65
Var Comp 6	47.90	0.06	11.70	1.50	0.59	0.31	0.24	35.43
Var Comp 7	46.95	0.10	10.32	0.08	2.29	1.41	0.24	33.65
Var Comp 8	28.30	4.01	18.53	3.39	2.17	1.15	0.94	33.20
Var Comp 9	40.55	4.55	6.44	3.36	2.86	1.77	1.15	26.75
Var Comp 10	35.25	6.69	6.77	5.23	2.98	2.25	0.22	25.05
PFS Comp.	33.45	0.65	6.99	0.78	7.21	6.11	0.43	24.13

Head sample analysis varied between 23% and 48% zinc, while iron assayed between 6% and 18% higher on all samples, when compared to the Kipushi 2016 PEA composite.

13.2.3 Gravity Separation Testwork

HLS and shaking table (ST) tests were conducted on the 11 composites using the flowsheet developed in the Kipushi 2016 PEA. The HLS results at a density cut point of 3.1 g/cm^3 are summarised in Table 13.5.

Table 13.5 Kipushi 2017 PFS HLS Tests Summary Results

Sample	Conc. Mass % of HLS Feed	HLS Conc. Grade			HLS Conc. Recovery		
		Zn (%)	Pb (%)	Fe (%)	Zn (%)	Pb (%)	Fe (%)
Var Comp 1	93.2	34.1	1.3	17.4	99.9	99.7	97.9
Var Comp 2	56.7	53.0	0.1	8.4	98.7	56.7	92.2
Var Comp 3	89.5	40.4	0.1	13.3	99.7	89.5	98.3
Var Comp 4	93.1	43.4	0.1	15.1	99.8	93.1	99.7
Var Comp 5	67.7	31.1	0.1	17.3	97.7	67.7	95.2
Var Comp 6	98.5	45.7	0.1	13.0	99.8	98.5	99.8
Var Comp 7	90.0	49.9	0.1	10.8	99.7	90.0	98.8
Var Comp 8	92.1	29.2	3.8	19.4	99.5	96.0	99.9
Var Comp 9	83.9	44.3	4.8	7.8	99.0	99.5	97.0
Var Comp 10	81.9	39.8	7.3	7.8	98.5	98.1	95.5
PFS Comp.	69.3	49.2	1.0	11.2	99.0	97.4	95.4

These results indicate that the HLS is highly effective in rejecting dolomite with zinc recovery in excess of 99% to sinks, however other heavy sulphide minerals associated with copper; lead; and iron, resulted in a concentrate that mostly did not meet the required product specifications. Shaking table tests also produced a poor concentrate specification and in line with feed composition as reported in Table 13.6.

Table 13.6 Kipushi 2017 PFS HLS Tests Summary Results

Sample	Conc. Mass % of ST Feed	ST Conc. Grade			ST Conc. Recovery		
		Zn (%)	Pb (%)	Fe (%)	Zn (%)	Pb (%)	Fe (%)
Var Comp 1	70.6	37.2	3.3	18.0	74.1	96.1	87.6
Var Comp 2	67.5	52.3	0.1	5.7	85.6	69.6	82.5
Var Comp 3	68.5	43.4	0.1	14.7	75.1	68.5	86.9
Var Comp 4	70.5	50.8	0.1	12.6	71.8	73.9	79.8
Var Comp 5	59.4	36.8	0.1	15.0	79.2	59.4	82.8
Var Comp 6	68.8	52.7	0.1	10.7	69.6	69.9	76.8
Var Comp 7	71.8	56.4	0.1	7.4	75.7	74.1	71.7
Var Comp 8	67.0	32.6	6.4	18.9	69.5	93.0	78.9
Var Comp 9	66.1	46.9	8.3	6.8	73.8	93.4	78.1
Var Comp 10	62.5	41.1	7.2	7.0	73.4	86.5	77.8
PFS Comp.	64.7	49.9	1.6	8.1	76.8	93.2	80.6

Variability simulations on the basis of the Kipushi 2016 PEA flowsheet were undertaken in METSIM® using the expected range of ROM mineralogical compositions over the LOM. These simulations, and the reported variability testwork results, confirmed that the Kipushi 2016 PEA circuit could not consistently produce zinc concentrate that meets required specification, because other heavy sulphide minerals associated with copper lead and iron also reported to concentrate. Furthermore, input from KICO suggested that a fine (µm), rather than coarse (mm) concentrate, was required by the custom smelters.

On the above basis, KICO undertook further testwork that incorporated a milling and flotation circuit using the PFS development composite, specifically to ensure a saleable zinc concentrate (P_{100} <500 µm and >52% Zn).

13.2.4 Comminution

Bond Rod Work Index (BRWi), Bond Ball Work Index (BBWi) and Bond Abrasion Index (Ai), were conducted at Mintek using 5 of Kipushi variability samples, to provide information for comminution circuit sizing for the Kipushi 2017 PFS. Results are summarised in Table 13.7.

Table 13.7 Kipushi Comminution Summary Results

Sample Designation	Hole ID	Bond Rod Work Index (kWh/t)	Bond Ball Work Index (kWh/t)	Bond Abrasion Index (g)	Rod to Ball Work Index ratio
Var Comp 3	KPU001	8.89	8.45	0.07	1.05
Var Comp 5	KPU062	13.40	9.41	0.17	1.42
Var Comp 6	KPU062	8.47	7.72	0.06	1.10
Var Comp 7	KPU071	7.40	7.81	0.03	0.95
Var Comp 10	KPU086	10.30	9.12	0.09	1.13
Design		10.9	9.2	0.11	

The BBWi results at a 106 µm limiting screen size, and the BRWi results at a 1.18 mm limiting screen size for all composites, can be classified as being of soft hardness, with respect to treatment within a ball and rod milling. These results were used as a basis for Kipushi 2017 PFS. It is relevant to note that these results are applicable to the RoM composite samples only and not the DMS concentrate samples, which form the basis of the mill design. Notwithstanding this, the differences are not expected to be large, and thus the use of these values was deemed appropriate for the Kipushi 2017 PFS.

13.2.5 Mineralogy

13.2.5.1 Bulk Mineralogy

A bulk mineralogical analysis PFS composite was conducted at Mintek. Results as compared with the Kipushi 2016 PEA composite analysis are detailed in Figure 13.3. Based on the samples analysed to date, it can be seen that the key minerals in order of abundance are: sphalerite; dolomite; pyrite; quartz, galena, and chalcopyrite. It can also be seen that the zinc and dolomite grades are largely inversely proportional to one another.

Figure 13.3 Bulk Modal Mineralogy Comparison

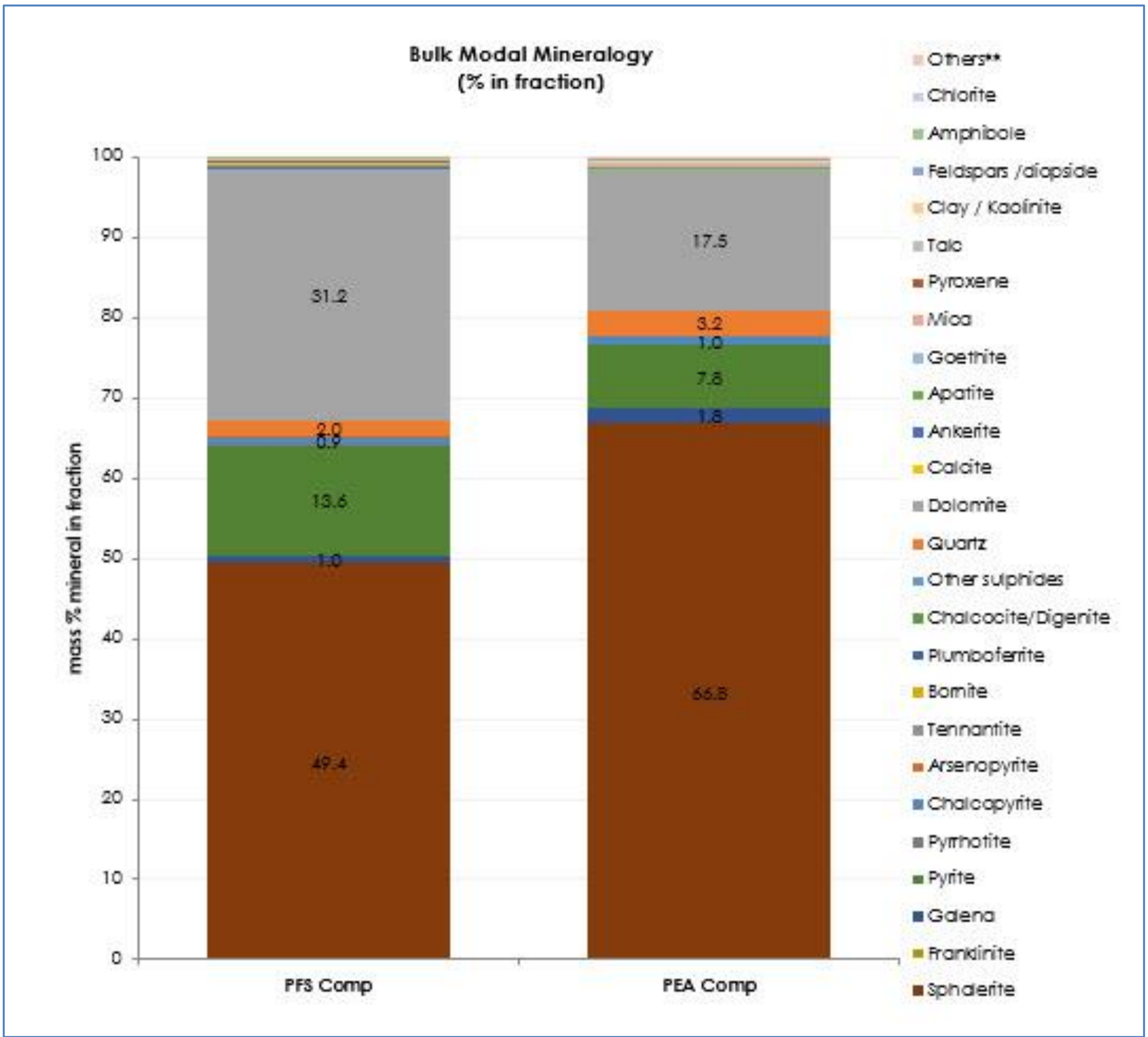


Image provided by Ivanhoe, 2018.

The PFS composite sample results confirmed that the Big Zinc is predominately sphalerite (49%) with chalcopyrite (1%) and galena (1%) as minor constituent, the main gangue minerals in order of abundance are dolomite (31%); pyrite (14%); quartz (2%).

In moving forward with the FS, knowledge of the Kipushi mineralogy will be further expanded, to understand how the LOM mineral variability will affect plant design, operation, and the product produced. It is noteworthy that the Kipushi 2016 PEA mine sample head grade is higher than any of the LOM average zinc grades, (circa 40% Zn) and with DMS and milling alone, the product will likely meet the required product specification without any subsequent flotation step.

In moving forward with the FS, knowledge of the Kipushi mineralogy will be further expanded upon, to understand how mineral variability over the LOM will affect plant design and operation, and the product produced. It is noteworthy that the Kipushi 2016 PEA mine sample head grade is higher than any of the LOM average zinc grades, (circa 40% Zn) and with DMS and milling alone, the product will likely meet the required product specification without any subsequent flotation step.

13.2.5.2 Mineral Grain Size Distributions

The grain size distributions analysis was also conducted for sphalerite, chalcopyrite, galena, and pyrite at 100% passing 1.7 mm and the results are presented in Figure 13.4.

Figure 13.4 Cumulative Mineral Grain Size Distribution

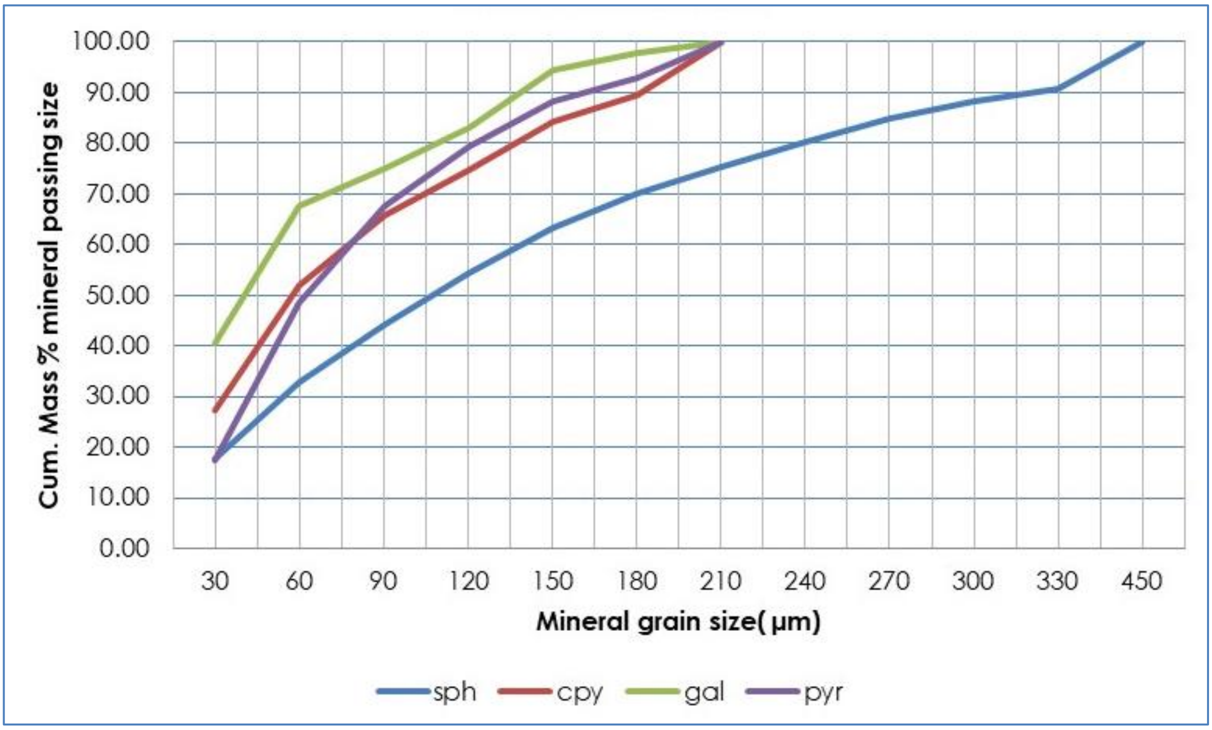


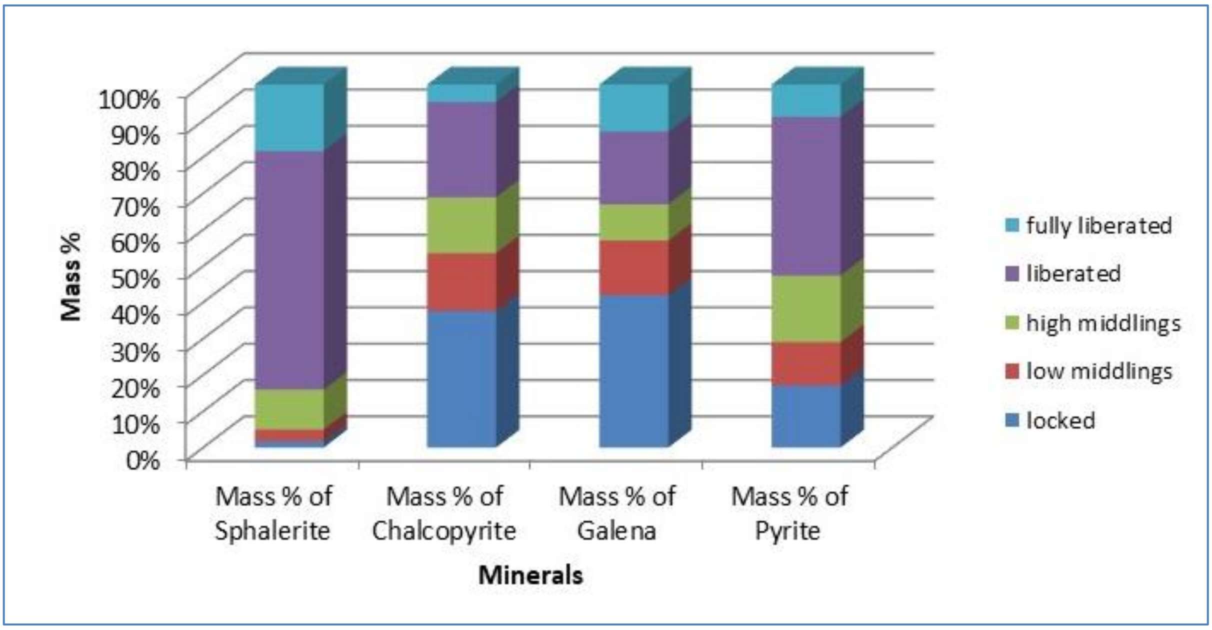
Figure provided by Ivanhoe, 2018.

Grainsize analysis showed that whilst sphalerite is relatively coarse grained with an average grain size of 105 µm, the other sulphide minerals largely have a grain size less than 120 µm. Galena and chalcopyrite are particularly fine grained an average grain size ~60 µm.

13.2.5.3 Liberation Analysis

The degree of liberation (P_{100} = -1.7 mm) for the four key minerals of interest (volume basis) is presented in Figure 13.5.

Figure 13.5 Liberation Analysis Curve



Courtesy of Mintek, 2017.

It is relevant to note that:

- 84.3% of the sphalerite is either highly or fully liberated; and whilst chalcopyrite and galena both have a similar degree of locked particles (~40%), copper recovery in the copper/lead circuit was poor, whilst lead recovery was very good.

13.2.5.4 Mineral Associations

The relative proportion of the main minerals associated with sphalerite, pyrite, galena and chalcopyrite is presented in Table 13.8. Mineral association data is derived from shared boundaries amongst the identified mineral grains. The higher the associated percentage is, the greater the degree of boundary-sharing between mineral species.

Table 13.8 Kipushi 2017 PFS Sample Mineral Association Summary Results

Associated Minerals	Sphalerite	Pyrite	Galena	Chalcopyrite
Free Surface	78.34	43.18	32.41	32.34
Sphalerite	0	30	18.47	29.42
Franklinite	0.21	0.02	0	0
Pyrite	10.92	0	33.78	19.37
Galena	0.51	2.58	0	0.57
Chalcopyrite	1.22	2.21	0.85	0
Arsenopyrite	0.09	0.78	0.6	3.19
Pyrrhotite	0.16	6.96	0.1	0.29
Other Sulphides	0.05	0.07	0.15	4.75
Mica	1.15	1.44	4.85	1.48
Quartz	1.19	1.56	3.2	0.72
Dolomite	0.59	2.44	0.31	0.27

The following should be noted regarding the presented results:

- The mineralogy analysis is based on a P_{100} of 1.7 mm.
- Both chalcopyrite and galena have a significant sphalerite association, thus possibly explaining the high zinc losses in the copper and lead circuit and possibly, the copper carry over to the zinc concentrate.
- Galena and chalcopyrite both have similar degrees of liberation, yet copper recovery in the lead/copper circuit is poor.

13.2.5.5 Electron Probe Microanalysis

Analysis of Mintek's microprobe work is presented in Table 13.9. This analysis is based on a 20 kV, 30 nA probe, with a spot size of 5 µm. MDM's analysis suggests an average sphalerite composition of (Zn0.975Fe0.025S).

Table 13.9 Electron Probe Microanalysis (Mintek, MDM, 2017)

	Mintek, Mass Analysis (%), n = 84 samples				Molar Analysis (%), (ZnFeS), n = 49 samples		
Element	**S**	**Fe**	**Zn**	**Total**	**S**	**Fe**	**Zn**
Min.	29.49	0.88	63.41	99.19	100	1.54	95.53
Max.	35.07	2.53	68.49	100.49	100	4.40	99.2
Avg.	32.77	1.47	65.73	99.96	**100**	**2.55**	**97.66**
Σ	0.83	0.38	0.92	0.20		0.7	1.0

13.2.6 Flotation Testwork

13.2.6.1 Background

Two alternate processing options were evaluated, the testwork results of which formed the basis for a conceptual techno-economic trade-off study conducted by MDM. The objective being to select a preferred process route to be further developed to the level of detail required to support a PFS.

For the ROM head sample, two alternate processing options were evaluated, namely:

- Option 1 - full stream ROM milling (P_{80} = 106 µm) followed by differential flotation.
- Option 2 - DMS pre-concentration followed by the milling (P_{80} = 106 µm) and differential flotation of the DMS concentrate and crusher circuit's fine fraction (-1 mm).

The differential flotation circuit is illustrated in Figure 13.6 below, tests were conducted using the flotation feed material as specified above and the feed composition detailed in Table 13.10. In the differential float, a copper/lead concentrate is first produced, followed by zinc flotation and pyrite depression in the subsequent flotation stage. The zinc rougher tails and the copper/lead concentrate are discarded as final tails.

Figure 13.6 Differential Flotation Circuit for PFS Composites

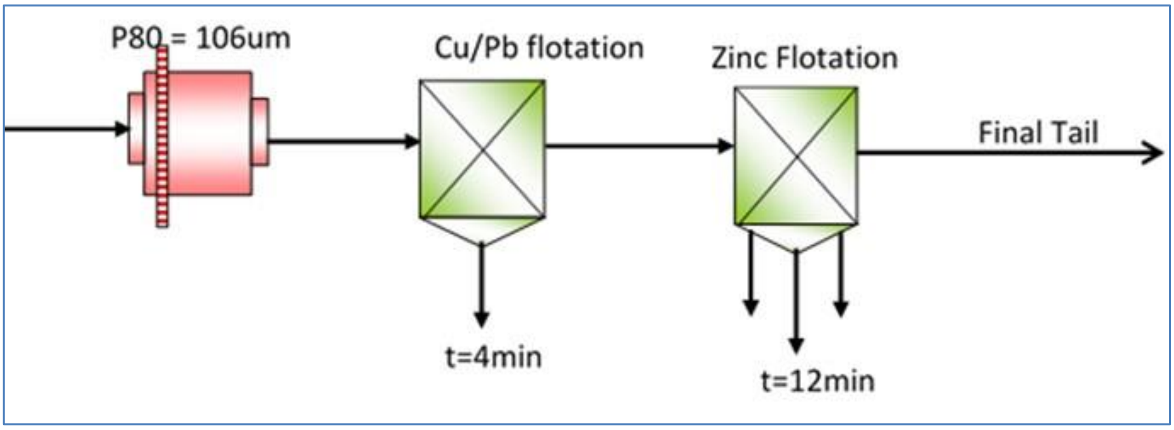


Table 13.10 Flotation Feed Composition for Options 1 and 2

	Units	Option 1 (ROM)	Option 2 (DMS)
Zinc (Zn)	%	32.7	43.7
Iron (Fe)	%	6.8	8.7
Sulphur (S)	%	24.5	32

The baseline reagent suite used for the flotation testwork is described below:

- Cu/Pb Conditioning (in Milling): Soda ash, zinc sulphate and sodium cyanide milled with the ore to a P_{80} = 106 µm at a pH=9.2.
- Cu/Pb float: SEX collector and MIBC.
- Zinc conditioner: Copper sulphate activator and lime for pH 11.5 correction.
- Zinc float: SIPX collector and MIBC.

13.2.6.2 Testwork Results

Option 1 (ROM Mill Float)

For the same conditions, duplicate flotation tests were conducted on the milled PFS composite head sample and the ROM Float tests are summarised in Table 13.11.

Table 13.11 ROM Sample Flotation Results Summary

Mintek Test #	Stream	Mass %	Grade				Recovery			
			Cu (%)	Pb (%)	Zn (%)	Fe (%)	Cu (%)	Pb (%)	Zn (%)	Fe (%)
Test 3	Cu-Pb Conc	14.9	0.9	4.1	8.6	10.8	32.2	93.0	3.7	22.5
	Zn Conc.	60.3	0.4	0.1	54.5	6.5	63.0	5.1	95.8	54.8
	Zn Tails	24.8	0.1	0.1	0.6	6.6	4.8	1.9	0.4	22.7
	Final Tails	39.7	0.4	1.6	3.6	8.2	37.0	94.9	4.2	45.2
Test 10	Cu-Pb Conc	11.0	1.4	6.3	11.4	16.4	17.6	92.0	3.7	25.2
	Zn Conc.	58.2	0.9	0.1	54.6	4.6	63.1	5.9	94.7	37.5
	Zn Tails	30.8	0.5	0.1	1.8	8.6	19.2	2.0	1.6	37.3
	Final Tails	41.8	0.8	1.7	4.3	10.7	36.9	94.1	5.3	62.5

The duplicate ROM float achieved a zinc grade of 54% Zn and recovery of 95%. The iron grade in final concentrate was in one test, below the desired 6% Fe and in the other, slightly above, but still below the 8% Fe grade for which toll penalties apply.

Option 2 (DMS Mill Float)

About 30 kg of PFS composite was subsampled and screened at -1 mm to prepare bulk HLS feed sample. The screen oversize (-20+1 mm) was subjected to a bulk HLS test in a bucket using a medium density of 3.1 g/cm³ to produce a concentrate sample for flotation testwork for Option 2. The HLS sinks was then combined with the screened -1 mm fines and send prepared for flotation testwork.

For the same test conditions as above, triplicates flotation tests were conducted on the DMS concentrate sample produced from the PFS composite sample. The results of the tests undertaken are presented in Table 13.12.

Table 13.12 DMS Conc Sample Flotation Results

Mintek Test #	Stream	Mass %	Grade				Recovery			
			Cu (%)	Pb (%)	Zn (%)	Fe (%)	Cu (%)	Pb (%)	Zn (%)	Fe (%)
Test 6	Cu-Pb Conc	17.8	1.3	5.6	17.0	19.0	39.8	95.2	6.9	34.5
	Zn Conc.	69.5	0.5	0.1	57.9	6.2	56.5	4.2	91.9	44.3
	Zn Tails	12.7	0.2	0.1	3.9	16.3	3.6	0.6	1.1	21.2
	Final Tails	30.5	0.8	3.3	11.5	17.9	43.5	95.8	8.1	55.7
Test 9	Cu-Pb Conc	16.8	1.7	6.1	14.4	18.7	46.6	94.6	5.5	35.3
	Zn Conc.	66.5	0.4	0.1	61.2	4.6	48.2	4.3	92.9	34.4
	Zn Tails	16.7	0.2	0.1	4.1	16.1	5.2	1.1	1.6	30.3
	Final Tails	33.5	0.9	3.1	9.2	17.4	51.8	95.7	7.1	65.6
Test 11	Cu-Pb Conc	17.4	1.0	5.7	17.2	19.5	30.1	94.9	6.5	38.3
	Zn Conc.	67.7	0.6	0.1	62.1	4.8	66.9	4.3	91.6	37.0
	Zn Tails	15.0	0.1	0.1	5.7	14.7	3.0	0.7	1.8	24.8
	Final Tails	32.3	0.6	3.1	11.9	17.3	33.1	95.7	8.4	63.0

The DMS concentrated sample achieved an average grade of 60% zinc and a recovery of 92% for the triplicates tests conducted. The DMS concentrate float circuit produced a higher-grade concentrate but had a lower overall zinc recovery compared to the straight ROM float circuit.

The performance of the proposed circuits from a recovery, grade and mass pull perspective for Option 1 and Option 2 are presented in Table 13.13.

Table 13.13 Circuit Performance Summary

	Option 1				Option 2 (DMS)			
	Overall Zinc Loss (%)	**Overall Mass Pull (%)**	**Circuit Mass Pull (%)**	**Zinc Conc. Grade (%)**	**Overall Zinc Loss (%)**	**Overall Mass Pull (%)**	**Circuit Mass Pull (%)**	**Zinc Conc. Grade (%)**
DMS circuit	N/A	N/A	N/A		2.6	26.2	30.0	
Cu/Pb flotation	3.8	14.9	14.9	8.6%	6.3	13.6	18.4	17.0
Zn flotation	0.5	60.3	70.9	**53.4%**	0.5	49.9	82.8	**61.0**
Total	**4.3**				**9.5**			

Note: Zinc losses are to tails.

The following points should be noted:

- Whilst gravity testwork in a heavy liquid solution (HLS) yielded zinc recoveries greater than 99%, these recoveries need to be moderated using Tromp curves to give real-life plant operating parameters for a DMS cyclone. For the process trade-off study undertaken, DMS recovery is of the order of 98%.
- In Option 2, the DMS option carries a 7.6% grade improvement over Option 1, but at the expense of 5.2% loss in zinc recovery.
- In Option 2, zinc losses in the copper/lead circuit are significantly higher than in Option 1. Grade / recovery relationships will be optimised in the FS testwork programme.

Based on the cost estimates prepared by MDM for both options, KICO decided that Option 2 gave the optimum techno-economic solution and the study progressed forward on this basis.

13.2.7 Testwork Representivity

The metallurgical testwork results for Kipushi 2017 PFS, and the corresponding head assays as reported, are in close alignment with the average weighted grades presented in Kipushi 2017 PFS mine plan. A summary of the testwork assays and the LOM grades reported on in the Mine Plan, are presented in Table 13.14. Except for iron, the base metal sulphides appear to be mostly representative of the weighted average values reported in the mine plan. The degree of iron substitution in sphalerite, will have some impact on the product grade achieved.

Table 13.14 Alignment between Kipushi 2017 PFS Mine Plan and Testwork

Elements	Kipushi 2017 PFS			Testwork		
	Min. Grade	Weighted Average Grade	Max. Grade	Applied by MDM	Measured	Calculated by Mintek
Zn %	22.94	32.14	36.04	32.60	33.45	34.10
Fe %	7.54	8.34	8.80	7.28	6.99	7.58
Pb %	0.35	0.85	1.38	0.81	0.65	0.85
Cu %	0.26	0.53	1.80	0.40	0.43	0.48
S %	19.36	23.74	26.80	24.03	24.13	24.64

13.2.8 PFS Testwork Summary by Section/Unit Operation

A PFS development composite sample with a LOM average head composition was subjected to a series of sequential metallurgical tests, where each test represents the natural progression of ore, through a series of plant sections/unit operations. Whilst the tests are batch in nature, the testwork was designed to enable a full mass and elemental balance, from the head sample to the final product and tails samples.

Mintek's raw data is summarised in Table 13.15.

The four unit operations/sections that form part of the testwork programme are described below:

- Crushing -20 mm material, deportment to -20 mm to + 1 mm and -1 mm fractions (one set of results).
- HLS test at chosen split density (3.1 t/m^3) (one set of results).
- Re-composition of the HLS concentrate and crusher fines to feed float plant.
- Differential flotation of chalcopyrite and lead and sphalerite and pyrite from DMS concentrate.

Table 13.15 Testwork Results by Section

	Plant Head Grade (-20 mm)	Feed (-20 to +1 mm)	Feed (-1 mm)	DMS Conc. (-20 to +1 mm)	DMS Tails. (-20 to +1 mm)	DMS Rec.
	%	%	%	%	%	%
Mass	100	87.2	12.8	60.4	26.8	
Zn	33.45	35.30	25.94	49.23	1.11	96.61
Pb	0.65	0.81	1.10	0.97	0.06	82.78
Fe	6.99	7.80	6.08	11.20	1.23	99.50
Ca	7.21	6.82	6.25	1.27	19.77	12.90
Cu	0.43	0.48	0.48	0.54	0.07	77.46
Mg	6.11	4.06	3.64	0.48	12.39	8.21
S	24.13	24.60	24.89	36.38	1.37	102.45
Si	0.78	0.45	0.80	0.95	1.01	145.93
	Recon. Float Feed	**Float Conc.**	**Float Tails**	**Float Rec.**	**Total Tails**	**Overall Rec.**
	%	%	%	%	%	%
Mass	73.2	49.8	23.4		50.2	
Zn	43.80	59.52	10.44	92.37	7.63	88.55
Pb	1.03	0.07	3.08	4.3	1.23	5.06
Fe	8.72	5.42	15.73	42.24	8.55	38.57
Ca	3.11	0.76	8.10	16.56	13.60	5.23
Cu	0.52	0.44	0.68	58.02	0.42	51.36
Mg	1.70	0.29	4.70	11.43	11.88	2.33
S	32.00	33.04	29.78	70.20	15.30	68.15

13.2.9 Testwork Analysis and Interpretation

The overall metal accountability by sequential unit operations is presented in Table 13.16.

The various laboratory tests were analysed, and the incremental sectional accountabilities were determined.

The zinc accountability varies between 97% and 105% between various sections of the operation as tested with 5% variability. As such these testwork results were determined acceptable for PFS requirements.

For the purposes of the overall plant product summary, the feed and product grade errors were reduced, and this calculated a final concentrate grade of 58.9% zinc, at 90.2% recovery.

Table 13.16 Summary of Accountability of Raw Data

Element	Crushed ore feed split	HLS test	Float feed recon.	Float test 6 in Zn con.	Float test 9 in Zn con.	Float test 11 in Zn con
Zn	102%	98%	97%	100%	100%	104.6%
Pb	130%	85%	104%	101%	104%	102%
Fe	108%	104%	85%	112%	102%	101%
Ca	94%	102%	145%	101%	103%	98%
Cu	112%	82%	99%	107%	115%	110%
Mg	66%	102%	164%	103%	96%	99%
S	102%	104%	93%	94%	85%	92%
Si	63%	215%				

13.3 Comments on Section 13

Sufficient testwork with a representative sample from the planned mining area were conducted to support the PFS and the overall circuit developed is robust, ensuring that a saleable concentrate specification should be met, based on the current annual production schedule.

The DMS plant alone was identified to have excellent dolomite discard capabilities (with minimal zinc losses). However, the high concentrate base metal recovery, reduced final zinc grade, and large -20 mm particle size was not ideal as a saleable product. This necessitated the inclusion of milling and flotation to the flowsheet, so as to consistently meet a saleable concentrate specification.

The PFS mineralogical and process testwork identified that milling produces a more favourable size product, and the associated flotation section has the ability to selectively discard 96% of the lead, 43% Copper albeit at the expense of a 6.3% loss in zinc.

Because iron levels in the feed affect concentrate grade, it is important to define the extent of iron variability at a more granular scale, as well as the level of iron substitution in the sphalerite matrix.

Although the DMS discards provides material for required mine backfill, it results in final tailings that are potentially acid generating, and thus the requirement for water treatment/ neutralisating before discharging into the environment.

Based on the Kipushi 2017 PFS, the ROM dolomite content (the primary gangue mineral), is expected to vary LOM between 29% and 48% on an annualised basis, and the Kipushi 2017 PFS plant design and the associated mass balance, is based on a dolomite content of 31%. The variability of dolomite content needs to be further studied in the next phase, to ensure that the circuit, especially downstream of the DMS, is designed to handle the variable feed streams.

14 MINERAL RESOURCE ESTIMATES

On behalf of KICO, the MSA Group (MSA) has completed a Mineral Resource estimate for the Kipushi Project (Kipushi).

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 2019 Resource Update, aside from those already mentioned in Section 4 of this report.

The Mineral Resource estimate incorporates drilling data collected by KICO from March 2014 until November 2015 inclusive and May 2017 to November 2017 inclusive, which, in the 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 RM software, together with Microsoft Excel, JMP, and Snowden Supervisor for data analysis. The Mineral Resource estimation was completed by Mr Jeremy Witley, the Qualified Person for the Mineral Resource.

14.1 Mineral Resource Estimation Database

The Mineral Resource estimate was based on geochemical analyses and density measurements obtained from the cores of diamond drillholes, which were completed by KICO from March 2014 to November 2015 and from May 2017 to November 2017, with the cut-off date for data included in this estimate being 26 April 2018. As at the cut-off-date, there were no outstanding data of relevance to this estimate and the database was complete. In addition to the KICO drillholes, 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 was captured in spreadsheets:

- Collar information;
- Downhole surveys where there are no survey data for a drillhole, the collar survey inclination and bearing were used as the downhole survey;
- Assays grades of Cu, Pb, Zn, S, Fe, As, and density;
- Lithological log;
- Mineralisation log.

Once the data were captured, the accuracy of the capturing was determined by checking 10% of the captured data against the hardcopy logs. The data were then checked for completeness to ensure that each drillhole record has corresponding records for collar, downhole survey, assay, lithology and mineralisation. Missing aspects of the data were sought and captured if found. The maximum depth of each drillhole was compared across each of the tables to identify whether logs were complete. Any discrepancies were checked and rectified where appropriate.

Once the check for completeness was complete, the integrity of the data was checked:

- The drillhole name was compared to the level, section and cubby number recorded in the collar table. Discrepancies were checked against hardcopy records and corrected where necessary.
- The dip of the drillhole is recorded in the drillhole name, this was compared to the dip from the survey sheets. Discrepancies were checked with the hardcopies and were corrected where necessary.

- Consistency in the drillhole name between tables was compared and where transcription errors or errors in the hard copy data were found, the drillhole names were modified appropriately.
- Duplicated logs were removed. Where duplicate data were found, the most complete sheet was used.
- Missing, duplicated or overlapping intervals were identified by summing the length of intervals within a specific hole, and comparing the sum to the depth in the collar table.
- The range of reported assays was checked to ensure that elements were consistently reported in percent or ppm as appropriate.

Once the data had passed the capturing validation tests it was imported into a Microsoft Access database for further checks. 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 from March 2014 to November 2015 and a further 59 from May 2017 to November 2017. The data from these holes are stored in a Microsoft Access database that in the Qualified Person's opinion conforms to modern acceptable database management protocols. The information contained in the database is comprehensive and contains data tables for collar surveys, downhole surveys, lithology, structure, geotechnical measurements and observations, sample assays and density.

Eight Gécamines drillholes were re-sampled by KICO. Infill sampling of these holes was also completed where Gécamines had not sampled the lower-grade intervals within the mineralised envelope. The original Gécamines data were replaced with the KICO re-sampled data for the purposes of the Mineral Resource estimate.

Eleven of the Gécamines holes were twin-drilled by KICO (Table 14.1). Where the holes were drilled within a few metres of one another, the Gécamines holes were discarded from the final database used for modelling. This was necessary as the KICO drillholes were more completely sampled in the lower-grade mineralisation than the Gécamines holes and thus any short-range discontinuities in the lower-grade mineralisation due to different sampling protocols were avoided.

Table 14.1 Kipushi Twinned Holes

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

The KICO sample assay database contains assay data for a number of elements as shown in Table 14.2.

Table 14.2 Assays in KICO Sample Database

Element	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 than the lower detection limit, the value was assigned a minus value in the database, equivalent to the lower detection limit of that element multiplied by negative 1 (i.e. -0.1). For estimation purposes, all negative assays were re-assigned a zero value.

14.2 Exploratory Analysis of the Raw Data

14.2.1 Validation of the Data

A final validation exercise was completed by the Qualified Person for the Mineral Resource. The validation process consisted of:

- Examining the sample assay, collar survey, downhole survey, and geology data, to ensure that the data was complete for all 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 mineralisation.
- Checks for "from-to" errors, to ensure that the sample data do not overlap one another, or that there are no unexplained gaps between samples.

The data validation exercise revealed the following:

- Below detection limit values were set to negative values in the database. All below detection limit assays were set to a value of zero for estimation purposes.
- There are intervals of Gécamines drill core that were not sampled or assayed. These intervals were set to zero grade, on the assumption that there was no visible mineralisation worth sampling and thus the core interval is barren. The Gécamines cores were selectively sampled, and samples were only taken when mineralisation was visibly determined to be above a threshold perceived to be economic at the time. For this reason, the assignment of zero grades to un-sampled intervals in the Gécamines database may be considered conservative, although this is the only reasonable option for the data.
- There are intervals of KICO drill core that were not sampled or assayed. These intervals were set to zero grade on the assumption that there was no visible mineralisation worth sampling and thus the core interval is barren. The KICO cores were mostly sampled throughout the length within the mineralised zones, and the assignation of zero grades to un-sampled intervals will not result in any biases. For KPU075, a large part of the mineralised intersection was not sampled, it being used for metallurgical studies. For this hole the assays were set to null ("-") values where there are no sample assay data available within the mineralised zone (as observed by the mineralisation log).
- Seven of the drillholes from the recent KICO drilling programme were drilled for metallurgical purposes and were not sampled. For these holes the assay values were set to null ("-").
- Several holes were drilled to investigate elevated zinc-copper mineralisation outside of the main zones of mineralisation and were not included in the estimation data.
- The assay data available for the Gécamines holes varies in completeness. If the copper value is blank, the assays for each element were set to zero, including copper. Where a sample has copper and/or zinc values, but other assays are missing, these were set to null, and the copper and/or zinc values were retained.

- Several of the KICO specific gravity measurements are outside of expected limits. Ten measurements are less than 2.1 g/cm³ and were set to a null value ("-") by MSA. 22 measurements are greater than 5.25 g/cm³ and were set to null values.
- There are no unresolved "from-to" errors in the database.
- The assay values in the database are within expected limits for the Kipushi mineralisation. This is with the exception of a single silver value of 27,600 ppm that was discarded, the next highest value being 4,240 ppm.
- There are no assays at the upper detection limit that were not sent for over-limit assays.

Drillholes were discarded from the Gécamines database for a number of reasons:

- There are eight cases where an entire Gécamines drillhole had intersected the mineralised zone and no assays were captured. In each of these cases the drillhole was rejected from the estimation database.
- Four Gécamines drillholes appear to be incorrectly coordinated, as they do not plot in the expected position relative to other holes and the Kipushi mineralised zones. These drillholes are 1132/18/V+6/-60/SE, which does not fit the mineralised zones, 1138/1/R+31/-70/SW which plots well within the Fault Zone footwall, 1138/1/R+31/-70/NW mineralised intercept plots well within the Série 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 mineralisation and so the data from each hole that contain excessive sample lengths (>4 m) were examined. The assays from these holes were flagged and not used for grade estimation if they appeared to be composited data. The composite sample hole data were used in the construction of the model to define the mineralisation extents but were not used in the estimation of the grade block model. In total, the assays from 131 Gécamines holes were not used for grade estimation.
- Fourteen Gécamines holes had been drilled along or close to the plane of the mineralisation either in dip or strike direction in the Série Récurrenté. These holes were not used for grade estimation but were used for defining the extents of the mineralisation.
- The position of the mineralised zones in one hole (1132/4/U+30/-90) did not compare well with the surrounding KICO holes and was discarded from the estimation database.
- Eleven Gécamines holes had been twin-drilled and were removed in favour of the KICO drillholes.

In total there are 134 KICO drillholes that have assays and intersected the targeted mineralised zones. 106 Gécamines drillholes were deemed acceptable for use in the grade interpolation process and an additional 144 Gécamines drillholes were included for the purpose of defining mineralisation limits.

The validated KICO and Gécamines data were combined for grade estimation. Consideration of the lack of certainty in the quality of the Gécamines data was made when classifying the Mineral Resource into the respective CIM categories of Measured, Indicated, and Inferred.

14.2.2 Statistics of the Sample Data

The Gécamines sample data were captured from scans of hard copy hand written and digital logs. Gécamines tended to use a variety of sample lengths, considerably longer than what would normally be used in modern practice. In addition, as the database contains composite sample lengths, a number of extreme sample lengths were reported from the database, with 4.4% of the sample lengths being greater than 10 m (Figure 14.1). The most frequent sample lengths are between 2 m and 5 m and 82.5% of the sample records have a length of less than or equal to 5 m long. As mentioned in Section 14.2.1, Gécamines drillholes that contain well mineralised sample lengths that are excessive were flagged in the estimation database. These holes were used in the construction of the grade shell to define the mineralisation extents but were not used in the estimation of the grade block model.

Figure 14.1 Histogram and Cumulative Frequency Plot of the Sample Length Data - Gécamines

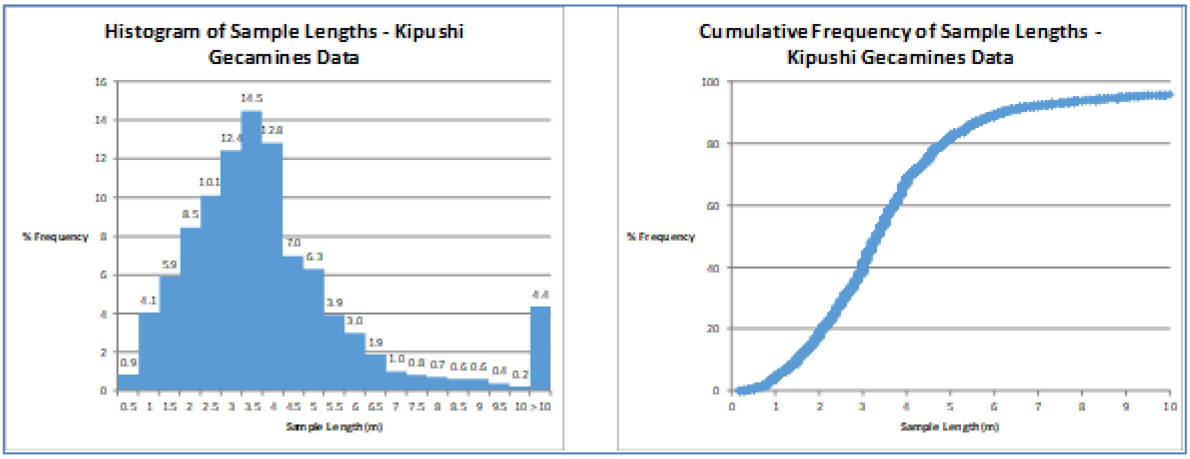


Figure by MSA, 2018.

The KICO sampling honoured the intensity of mineralisation and geological contacts. In homogenous zones nominal sample lengths of 1 m or 2 m were taken, with the longer samples tending to be taken from low-grade or waste zones (Figure 14.2).

Figure 14.2 Histogram and Cumulative Frequency Plot of the Sample Length Data – KICO

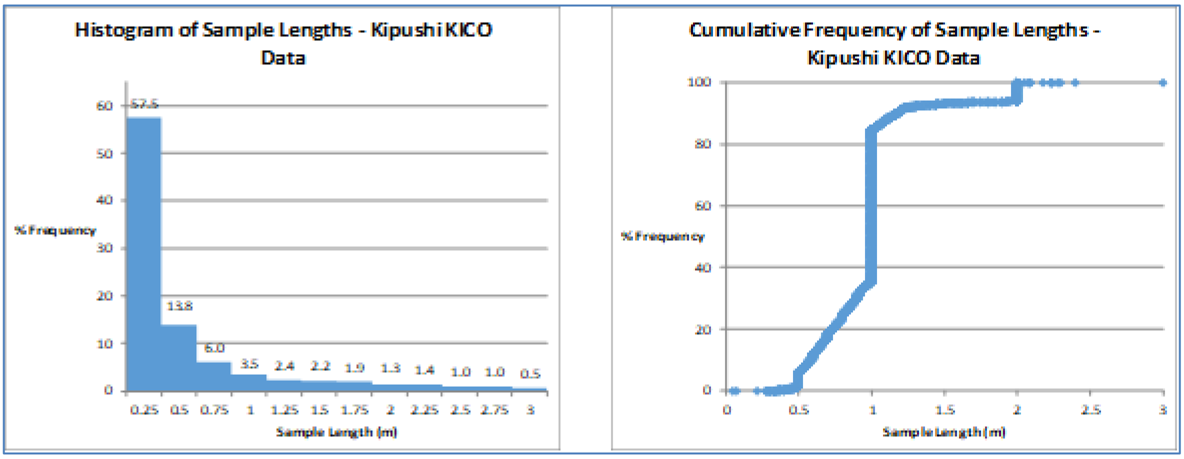


Figure by MSA, 2018.

14.2.3 Statistics of the Assay Data

Platinum and palladium assays are of negligible grade, these assays being largely below the detection limit with rare instances of assays of 20 ppb, 40 ppb, or 60 ppb. The assays for gold are low and only 11 values are greater than 0.5 g/t and there are only 41 values above 0.2 g/t. Two samples returned assays of 2.72 g/t and 3.16 g/t Au respectively. Samples from drillholes completed in 2017 were not assayed for platinum, palladium and gold.

Not all of the KICO samples were assayed for nickel, vanadium or uranium. The earlier drillholes completed by KICO were assayed for nickel and vanadium, but due to the low values experienced, they were discontinued from KPU030 onwards. KPU001 and KPU002 were not assayed for uranium.

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

As the assays for Pt, Pd, Au, Ni, and V are of negligible grade, these elements were not considered further in the Mineral Resource estimate.

The KICO samples were also assayed for mercury, uranium, molybdenum and rhenium. Some of the samples have significant grades for these elements, but overall they are low. Mercury assays are less than 200 ppm. 63% of the molybdenum assays are below the lower detection limit (5 ppm), and only 16 values are above 1,000 ppm. 71% of the rhenium assays are below the lower detection limit (0.10 ppm) and only eight values are greater than 50 ppm. Uranium values are generally low with approximately 98% of the values being below 10 ppm, 29 values being above 50 ppm and the maximum assay being 513 ppm. Given the low numbers of significant assays for Hg, Mo, and Re these elements were not considered further in the Mineral Resource estimate, as the value that they could contribute to the project is insignificant. Uranium may be considered a nuisance or deleterious element in situations where it exists in amounts too low to derive economic value. It is uncertain whether the amount of uranium at Kipushi will be of any impact to the project given the generally low values.

Copper, lead zinc, sulphur, arsenic silver, germanium, cobalt, cadmium, iron and density were considered of importance to the Kipushi Project. As a result, these were examined in greater detail and estimated into the Mineral Resource block model.

14.2.3.1 Univariate Analysis

A summary of the sample assay statistics of the un-composited data at Kipushi is shown in Table 14.3 for the Gécamines data and Table 14.4 for the KICO data.

Table 14.3 Summary of the Raw Validated Sample Data*[1] for the Gécamines Drillholes

Variable	Number of Assays	Mean Value	Minimum Value	Maximum Value
Cu%	2,182	2.41	0.005	60.80
Pb%	2,178	0.52	0.005	16.40
Zn%	2,182	9.91	0.005	63.15
S%	1,926	12.84	0.03	43.65
As%	1,823	0.17	0.005	7.46
Ag g/t	No Data	–	–	–
Ge g/t	No Data	–	–	–
Co ppm	No data	–	–	–
Cd ppm	No Data	–	–	–
Fe%	1,920	8.29	0.78	39.01

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

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

Variable	Number of Assays	Mean Value	Minimum Value	Maximum Value
Cu%	11,064	1.11	0.00	40.40
Pb%	11,064	0.17	0.00	17.90
Zn%	11,064	12.11	0.00	65.20
S%	11,064	12.1	0.0	51.70
As%	11,064	0.18	0.00	14.70
Ag g/t (ICPMS)	11,063	13.0	0.0	4,240
Ge g/t	11,064	33.7	0.0	19,600
Co ppm	11,064	38.4	0.0	25,300
Cd ppm	11,064	688	0	14,500
Fe%	10,886	6.49	0.19	51.90
Density g/cm^3	10,402	3.28	2.03	5.22

The Gécamines database does not contain values for silver, germanium, copper or cadmium as well as some copper, lead, zinc, sulphur, iron and arsenic values. The mean assay values for the KICO copper and lead data are less than those of the Gécamines data as the KICO cores were completely sampled in the potentially mineralised zones, unlike the Gécamines sampling that was selective aimed at higher grade copper or zinc mineralisation.

Several zones of mineralisation have been identified by Gécamines and KICO. The zones of mineralisation are either copper dominant or zinc dominant with varying amounts of other elements. The grade distributions are characterised by large amounts of low-grade data (below approximately 0.2% for copper and 5% for zinc), medium grade data and high-grade (above approximately 20% for copper and 20% for zinc) data.

Bivariate Analysis

Scatterplots were made that compare the grades of individual elements against one another. The scatterplots for the total data show various relationships that indicate mixed mineralisation domains. Several mineralisation styles at Kipushi exist, the zinc-rich zones resulting in different bivariate relationships than the copper-rich zones. No clear relationships were found between copper, lead, zinc, and cobalt. Mixed linear relationships are evident between copper and 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 14.3). 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 14.3 Sulphur and Cadmium Regressions

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

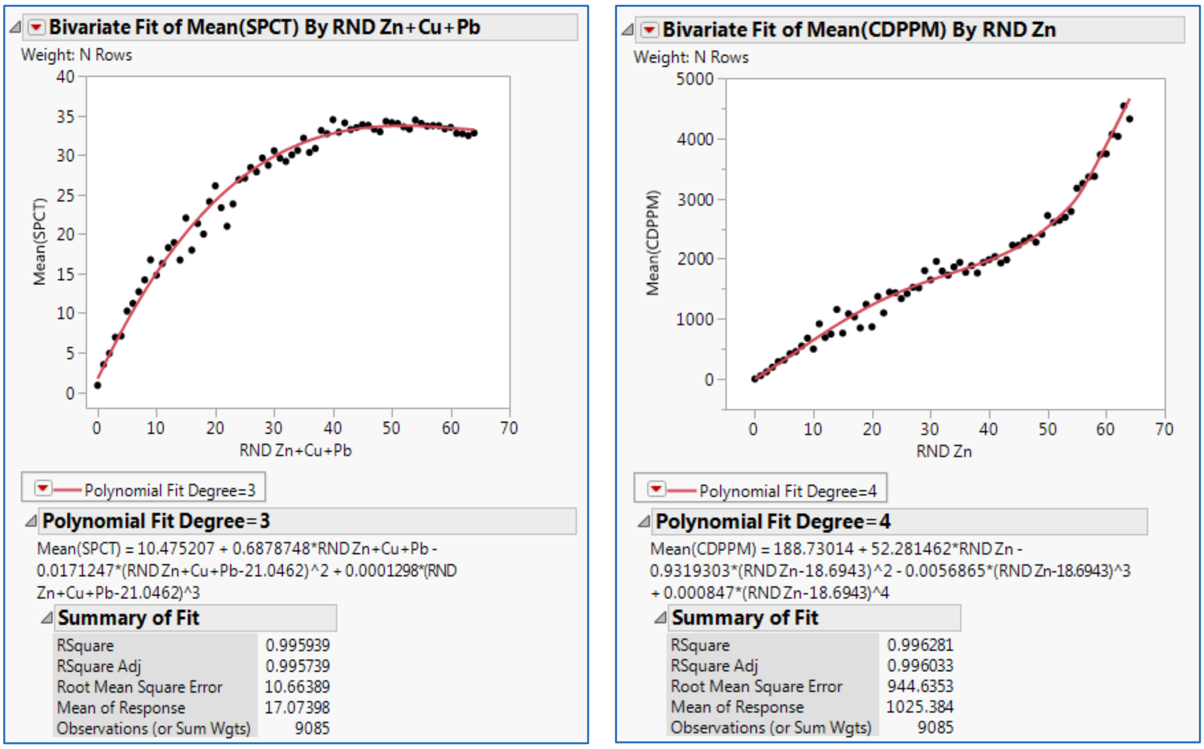


Figure by MSA, 2018.

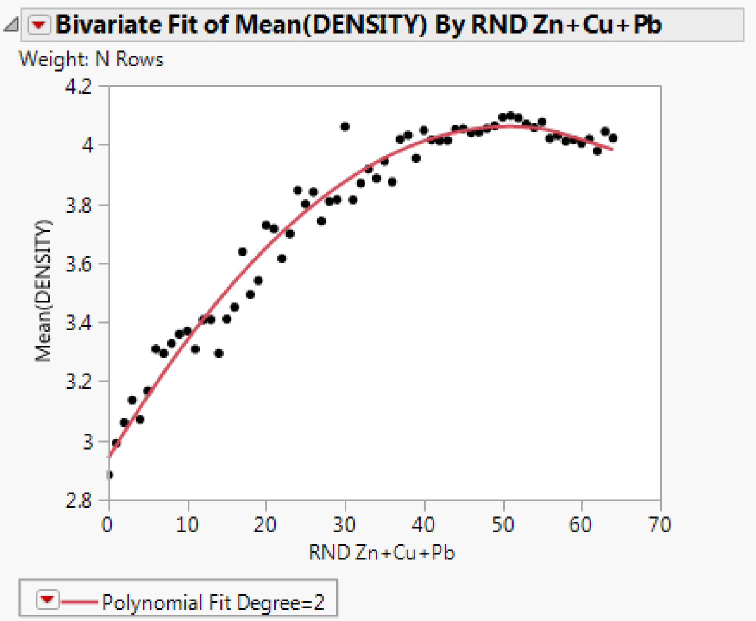
Density Determination

Density was measured by KICO on whole lengths of half core samples, using Archimedes principal of weight in air versus weight in water. Not all the KICO samples were measured for density. Many of the Gécamines density values were derived from a calculation or considered unreliable and so the Gécamines density values were discarded. A regression was formulated from the KICO measurements, in order to estimate the density of each sample based on its grade. This formula was applied to all the Gécamines samples, and to the KICO samples that did not have density measurements performed on them. It was found that a summation of copper, zinc and lead grade versus density produced a reasonable regression for the multi-element mineralisation at Kipushi, however the mineralisation at Kipushi is complex and it was difficult to produce a perfect fit for all grade ranges.

A second order polynomial curve was fitted to the data as shown in Figure 14.4, 52% Cu+Zn+Pb a slight decrease in density was observed with increasing grade.

It should be noted that use of regression formulae is not ideal and local biases will occur, however it is expected that on average the density for each zone will be accurate.

Figure 14.4 Density Regression



Polynomial Fit Degree=2

Mean(DENSITY) = 3.1346125 + 0.02587*RND Zn+Cu+Pb - 0.0004374*(RND Zn+Cu+Pb-21.0437)^2

Summary of Fit

RSquare	0.99039
RSquare Adj	0.99008
Root Mean Square Error	0.569095
Mean of Response	3.457039
Observations (or Sum Wgts)	9084

Figure by MSA, 2018.

14.2.4 Summary of the Exploratory Analysis of the Raw Dataset

- KICO assays below the detection limit were assigned zero values, they 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 mineralised intersection was not sampled, it being used for metallurgical studies. For this hole the assays were set to null values where there are no sample assay data available within the mineralised zone as defined by the mineralisation log. Seven holes were drilled specifically for metallurgical test-work. These were also assigned null values.
- The assay data available for the Gécamines holes varies in completeness. If the copper value is blank, the assays for each element were set to zero including copper. Where a sample has copper and/or zinc values, but other assays are missing, the other values were set to null and the copper and/or zinc values were retained. This is based on the assumption that the missing values were not assayed and assigning zero value to them would be incorrect.
- Drillholes were discarded from the Gécamines database for a number of reasons, such as no assays captured, incorrect coordinates, excessive samples lengths due to composite data being captured, and inappropriate drilling directions. Gécamines holes that had been twin-drilled by KICO were also removed from the estimation data set.
- In total, there are 134 KICO drillholes that have sampling data. 106 Gécamines drillholes were deemed acceptable for use in the grade interpolation process and an additional 144 Gécamines drillholes were included for the purpose of defining mineralisation limits.
- The quality of the Gécamines data is less certain than for the KICO data. Consideration of this was made when classifying the Mineral Resource into the respective CIM categories of Measured, Indicated, and Inferred.
- Copper, lead zinc, sulphur, arsenic silver, germanium, cobalt, cadmium, and density are considered of importance to the Kipushi Project. A number of other elements were assayed by KICO; however, their concentrations are not significant. Uranium may be considered a nuisance or deleterious element in situations where it exists in amounts too low to derive economic value. It is uncertain whether the amount of uranium at Kipushi will impact the project at the low-grades in which it occurs.
- Missing values for 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 mineralisation have been identified, either copper-rich or zinc-rich. These are spatially separate and need to be considered as separate domains in estimation.

14.3 Geological Modelling

14.3.1 Mineralised Zones

The mineralisation at Kipushi comprises 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 mineralisation occur, the Big Zinc and the Southern Zinc, which lie adjacent to the copper-rich Fault Zone mineralisation. In places, the Big Zinc mineralisation is juxtaposed against the Fault Zone, although in many areas zones devoid of significant mineralisation occur between them. A zone of high grade copper, silver and germanium occurs within the Big Zinc. The Southern Zinc zone is an elongate lense of sphalerite rich mineralisation parallel and juxtaposed against the Fault Zone mineralisation. The Southern Zinc becomes copper-rich and zinc-poor towards the south.

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

In total, ten zones were modelled as separate wireframes:

- Fault Zone - Zone 1. A low-grade zone (Zone 9) was defined within the Fault Zone.
- Big Zinc - Zone 2.
- Southern Zinc - Zone 3. A low zinc, moderate copper zone (Zone10) was defined towards the south.
- Série Récurrenté - Zone 4.
- Massive sulphide lense adjacent the Série Récurrenté - Zone 5. A massive sulphide lense occurs within it (Zone 8).
- High-grade copper zone within the Big Zinc - Zone 6.
- Fault Zone Splay - the high zinc-copper-germanium splay from the Fault Zone - Zone 7.

Mineralised zones were identified using a threshold value of 5% for zinc and 1.0% for copper. Strings were constructed along sections perpendicular to the dip of the mineralisation by snapping to the drillhole intercepts. The sections were examined along strike to ensure that the thickness trends of the mineralisation were continued from one section to the next. The interpreted strings were then linked to form wireframe solids.

All available validated data were used for the construction of the mineralised models. The Gécamines drillholes that were rejected from the grade estimation due to excessive sample lengths were also used.

The resulting wireframe shells show local irregularities although clear trends are evident, particularly for the Big Zinc that plunges steeply to the south-west. An isometric view of the wireframe models is shown in Figure 14.5.

Figure 14.5 Isometric View of Kipushi Wireframes and Drillholes (view is approximately to the north-west)

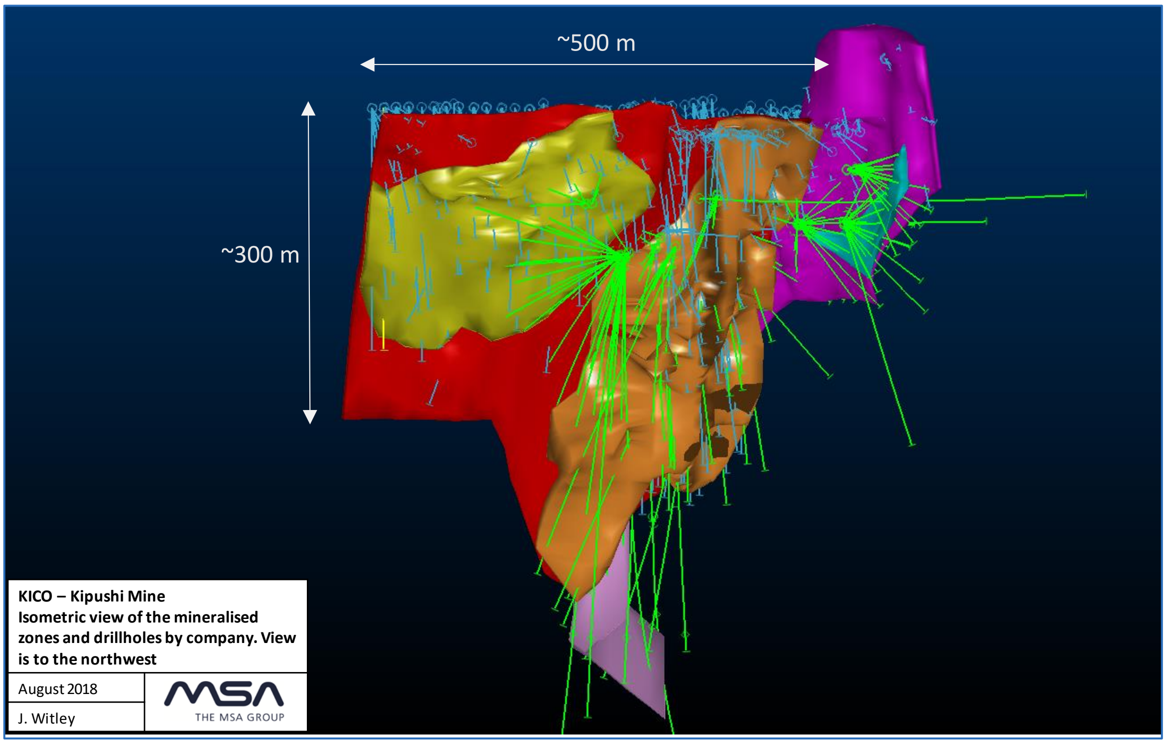


Figure by MSA, 2018.
Red Wireframe = Fault Zone (Zone 1).
Orange Wireframe = Big Zinc (Zone 2).
Yellow Wireframe = Southern Zinc (Zone 3).
Magenta Wireframe = Série Récurrenté (Zone 4).
Cyan Wireframe = High-grade Copper Zone Adjacent to Série Récurrenté (Zone 5)
Pink Wireframe = Fault Zone Splay (Zone 7).
Blue traces =Gécamines drillholes.
Green traces = KICO drillholes.

14.4 Statistical Analysis of the Composite Data

The drillhole sample data that were considered suitable for estimation purposes were selected by zone using the modelled wireframes and then composited to 2 m lengths using density-length weighting. The composites were de-clustered to a cell size of 20 mX, 20 mY, and 20 mZ by weighting by the number of data in each cell and summary statistics were compiled for each mineralised zone (Table 14.5).

The summary statistics were interrogated, paying particular attention to the variability (as exhibited by the coefficient of variation (CV)) and the skewness, as high skewness tends to be an indication of a number of particularly high-grade values within a generally lower-grade distribution.

Table 14.5 Summary Statistics (de-clustered) of the Estimation 2 m Composite Data for Grades and SG

Variable	Number of composites	Minimum	Maximum	Mean	CV	Skewness
Zone 1 – Fault Zone						
Cu %	601	0.00	42.25	3.53	1.23	3.1
Pb %	601	0.00	3.72	0.13	3.25	5.9
Zn %	601	0.00	43.83	3.77	1.80	3.2
S %	601	0.00	50.01	14.66	0.76	0.7
As %	482	0.00	9.33	0.45	1.89	4.7
Ag g/t	280	0.0	165.8	20.6	1.33	3.0
Ge g/t	280	0.0	433.7	35.5	1.62	4.5
Co ppm	280	0	13,121	243	5.02	7.6
Cd ppm	601	0	6,776	273	1.97	5.1
Density	617	2.69	4.67	3.24	0.10	1.2
Iron %	503	0.00	44.45	14.1	0.75	0.76
Zone 2 – Big Zinc						
Cu %	2,913	0.00	60.80	0.97	3.23	8.2
Pb %	2,913	0.00	16.77	0.80	2.75	3.8
Zn %	2,913	0.00	63.43	31.32	0.63	-0.3
S %	2,913	0.00	45.72	25.03	0.50	-0.9
As %	2,878	0.00	5.54	0.16	2.09	8.9
Ag g/t	2,105	0.0	196.1	14.7	1.33	3.3
Ge g/t	2,105	0.0	655.8	52.5	1.04	4.3
Co ppm	2,105	0	5,483	23	8.4	23.1
Cd ppm	2,913	0	5,557	1,543	0.72	0.4

Variable	Number of composites	Minimum	Maximum	Mean	CV	Skewness
Density	3,148	2.19	4.82	3.73	0.12	-0.7
Iron (%)	2,850	0.00	40.66	9.33	0.70	1.0
Zone 3 – Southern Zinc						
Cu %	217	0.00	28.71	2.95	1.34	2.8
Pb %	217	0.00	9.72	1.47	1.35	1.8
Zn %	217	0.00	61.7	24.31	0.67	0.05
S %	217	0.00	40.50	24.92	0.44	-1.0
As %	128	0.00	2.51	0.37	1.12	3.1
Ag g/t	103	1.6	3,106	131.1	2.96	5.5
Ge g/t	103	0.0	12,704.9	442.1	3.71	6.0
Co ppm	103	0	110	7	1.91	4.8
Cd ppm	217	0	14,273	2,880	1.20	1.7
Density	230	2.84	4.59	3.73	0.11	-0.5
Iron %	127	0.00	36.80	12.05	0.67	0.7
Zone 4 – Série Récurrenté						
Cu %	1,453	0.00	26.75	1.96	1.39	4.4
Pb %	1,453	0.00	1.94	0.03	5.18	9.7
Zn %	1,453	0.00	32.14	0.65	3.96	7.3
S %	1,453	0.00	35.61	2.82	1.66	4.0
As %	1,419	0.00	1.70	0.06	2.30	6.7
Ag g/t	583	0.0	96.8	7.9	1.05	3.4
Ge g/t	583	0.0	9.1	0.6	2.62	2.9
Co ppm	583	0	1,366	36	2.52	8.0
Cd ppm	1,453	0	1,714	41	3.75	6.7
Density	1,453	2.39	4.05	3.02	0.06	2.8
Iron %	1,453	0.00	32.89	3.64	0.96	4.3
Zone 5 – Série Récurrenté Footwall						
Cu %	78	0.18	5.40	1.78	0.63	1.6
Pb %	78	0.00	11.26	0.12	8.10	10.5
Zn %	78	0.00	54.29	5.02	2.67	2.7
S %	78	0.47	29.33	4.76	1.65	2.4
As %	78	0.00	0.69	0.08	1.62	3.7
Ag g/t	78	0.0	60.4	11.5	1.08	2.3

Variable	Number of composites	Minimum	Maximum	Mean	CV	Skewness
Ge g/t	78	0.0	28.5	3.16	2.10	2.3
Co ppm	78	0	1,229	30	3.85	9.6
Cd ppm	78	0	3,722	311	2.77	2.9
Density	82	2.82	4.07	3.08	0.10	2.2
Iron %	78	0.76	16.05	2.56	0.98	4.6
Zone 6 – High-grade copper zone within Big Zinc						
Cu %	115	0.88	32.53	6.64	0.85	1.7
Pb %	115	0.00	13.10	1.03	2.26	3.7
Zn %	115	0.01	54.90	26.86	0.67	-0.2
S %	115	1.20	42.05	26.24	0.45	-1.0
As %	115	0.01	0.86	0.19	0.89	2.1
Ag g/t	89	7.6	2,036.7	175.4	2.0	3.2
Ge g/t	89	5.7	410.0	75.0	0.92	3.2
Co ppm	89	0	4,577	182	2.76	6.4
Cd ppm	115	0	3,724	1,627	0.70	0.0
Density	138	2.67	4.79	3.81	0.12	-1.0
Iron (%)	115	1.19	31.45	11.20	0.60	0.8
Zone 7 – Fault Zone Splay						
Cu %	94	0.00	20.16	2.88	1.43	2.1
Pb %	94	0.00	0.09	0.01	1.88	3.1
Zn %	94	0.01	64.27	27.86	0.98	0.1
S %	94	0.48	38.83	26.49	0.39	-1.4
As %	94	0.00	12.43	2.26	1.45	1.5
Ag g/t	94	0.1	82.3	14.8	1.00	2.1
Ge g/t	94	0.0	599.8	144.3	1.13	0.8
Co ppm	94	0	2,210	98	2.36	7.1
Cd ppm	94	0	6,189	2,007	1.03	0.4
Density	94	2.86	4.63	3.76	0.12	-0.7
Iron %	94	0.55	40.35	12.74	0.98	0.8
Zone 8 – Massive Sulphide in Série Récurrenté Footwall						
Cu %	70	0.63	35.45	14.56	0.62	0.5
Pb %	70	0.00	5.42	0.21	3.70	4.8
Zn %	70	0.00	52.96	8.28	1.78	1.8

Variable	Number of composites	Minimum	Maximum	Mean	CV	Skewness
S %	70	0.59	31.67	19.12	0.47	-0.4
As %	70	0.00	5.35	0.58	1.95	3.3
Ag g/t	70	0.0	1,205.8	135.5	1.69	3.3
Ge g/t	70	0.0	67.5	16.0	1.04	1.3
Co ppm	70	0	5,182	259	3.14	5.5
Cd ppm	70	0	4,319	542	1.85	2.0
Density	73	2.87	4.20	3.53	0.10	-0.2
Iron %	70	0.96	27.26	13.49	0.55	0.2
Zone 9 – Low-grade zone in Fault Zone						
Cu %	37	0.00	4.87	0.56	1.37	4.1
Pb %	37	0.00	3.04	0.21	2.85	4.0
Zn %	37	0.00	42.79	7.12	1.32	2.7
S %	37	0.00	39.26	13.37	0.82	0.9
As %	34	0.00	2.05	0.26	1.95	3.1
Ag g/t	11	2.7	17.3	6.7	0.63	1.3
Ge g/t	11	5.0	38.9	15.5	0.63	1.1
Co ppm	11	0	24	12	0.72	-0.2
Cd ppm	37	0	2,536	491	1.11	2.1
Density	37	2.83	4.14	3.23	0.10	1.2
Iron %	34	0.00	34.84	10.69	0.81	1.2
Zone 10 – Southern portion of Southern Zinc						
Cu %	55	0.00	6.78	1.89	0.75	1.7
Pb %	55	0.00	0.51	0.05	1.88	3.3
Zn %	55	0.00	14.45	1.57	1.58	3.2
S %	55	0.00	40.46	8.51	1.24	2.4
As %	3	0.00	0.00	0.00	-	-
Ag g/t	0	-	-	-	-	-
Ge g/t	0	-	-	-	-	-
Co ppm	0	-	-	-	-	-
Cd ppm	55	0	912	101	155	3.2
Density	55	2.94	3.49	3.08	0.04	1.5
Iron %	19	0.00	34.35	9.16	1.45	1.4

For each element in most domains there are a significant number of composites with zero grade. These largely represent un-sampled intervals within the mineralisation wireframes, many of which are derived from Gécamines sample data for which sampling was selective. There are no silver, germanium and cobalt data available for the southern portion of the Southern Zinc (Zone 10), this zone being informed only by Gécamines data.

The copper distributions are generally characterised by moderate 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. have 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 dominantly 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 is 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 0.06% and 0.58% except for the Fault Zone Splay (Zone 7), which is high in arsenic and the mean arsenic grade is 2.26%.

The silver distributions have moderate CVs and strong skewness as a result of a small number of extremely high values. Mean silver grades are particularly high in the massive chalcopyrite rich zones (Zones 6 and 8).

Germanium CVs are low and distributions are moderately positively skewed except for Zone 4 and 5 that are generally of low germanium grade with a few values significantly higher than the mean value. Mean germanium values are high in the Big Zinc and the massive chalcopyrite and bornite rich zone (Zone 6) within the Big Zinc. Particularly high germanium values occur in the Fault Zone Splay (Zone 7) and the Southern Zinc contains a number of very high germanium grade samples (>1,000 g/t).

Cobalt distributions are positively skewed with high CVs caused by a small number of high values within a generally low-grade population.

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

The moderate CVs indicate that a linear method, such as ordinary kriging, is appropriate to estimate the grades. The zones with high CV's and that are strongly positively skewed are a result of a small number of high-grade values that can be considered outliers and measures that control their impact are required.

14.4.1 Outlier Control

The log probability plots and histograms of the composite data were examined for outlier values that have a low probability of re-occurrence, particularly where a small proportion of high-grade data makes up a disproportional amount of the domain mean, populations with high CVs and histograms with long tails. The outlier values identified were capped to a threshold as shown in Table 14.6. The threshold was set at the next highest value below the lowest identified outlier value. Decisions on the capping threshold were guided by breaks in the cumulative log probability plots and the location of the high-grade samples with respect to other high-grade samples.

The capping reduced the extreme CVs but several remained high (>2) in Zone 2 and Zone 4 where the distributions exhibit high skewness.

The lead, arsenic, silver, germanium and cobalt distributions are characterised by small numbers of high-grade values within dominantly low-grade populations. The high-grades tend to occur in clusters. In order to retain the high-grade values locally, without smearing of the values throughout their respective estimation domains, a restricted omnidirectional search of 7 m was applied on the data during interpolation without capping applied. This allows the high-grades to influence only the block in which they occur and the immediately surrounding blocks. The estimates using the uncapped data replaced the estimates using the capped data. The parameters used for the restricted search are described in Section 14.7.2.

Table 14.6 Values Capped and Their Impact on Sample Mean and CV

Attribute	Before Capping			After Capping			
	Number of Composites	Mean	CV	Cap Value	Number of Composites Capped	Mean	CV
Zone 1 – Fault Zone							
Cu %	601	3.53	1.23	24.3	2	3.51	1.18
Pb g/t	601	0.13	3.25	0.56	31	0.08	1.92
Zn %	601	3.77	1.80	28.0	14	3.59	1.66
As %	482	0.45	1.89	3.05	10	0.41	1.54
Ge g/t	280	35.5	1.62	116	5	29.7	1.06
Co ppm	280	243	5.02	323	21	62	1.40
Cd ppm	601	273	2.0	1938	11	255	1.64
Zone 2 – Big Zinc							
Cu %	2,913	0.97	3.34	9.93	56	0.78	2.39
Pb %	2,913	0.80	2.75	9.97	39	0.76	2.60
Ag g/t	2,878	0.16	2.09	1.58	29	0.15	1.44
Ge g/t	2,105	52.5	1.04	280	12	51.3	0.90
Co ppm	2,105	23	8.41	104	59	11	2.02

Attribute	Before Capping			After Capping			
	Number of Composites	Mean	CV	Cap Value	Number of Composites Capped	Mean	CV
Zone 3 – Southern Zinc							
Cu %	217	2.95	1.34	16.6	5	2.87	1.25
As %	128	0.37	1.12	0.95	5	0.33	0.80
Ag g/t	103	131.1	2.96	171	16	56.0	0.96
Ge g/t	103	442.1	3.71	813	14	160.3	1.49
Co ppm	103	7	1.91	20	6	5	1.24
Zone 4 – Série Récurrenté							
Cu %	1,453	1.96	1.39	17.2	8	1.93	1.31
Pb g/t	1,453	0.03	5.18	0.19	35	0.01	2.40
Zn %	1,453	0.65	3.96	3.85	67	0.37	2.41
As %	1,419	0.06	2.30	0.34	30	0.05	1.48
Ag g/t	583	7.9	1.05	40	6	7.8	0.95
Ge g/t	583	0.6	2.62	5.5	10	0.6	2.50
Co ppm	583	36	2.52	181	15	28	1.37
Cd ppm	1,453	41	3.75	240	67	24	2.34
Zone 5 – Série Récurrenté Footwall							
Pb g/t	78	0.12	8.10	0.011	5	0.002	2.02
As %	78	0.08	1.62	0.26	2	0.06	1.10
Ag g/t	78	11.5	1.08	40	2	10.9	0.95
Co ppm	78	30	3.85	134	3	20	1.60
Fe %	78	2.56	0.98	5.05	1	2.25	0.48
Zone 6 – High-grade copper zone within Big Zinc							
Pb %	115	1.03	2.26	4.15	8	0.75	1.66
Ag g/t	89	175.4	2.00	281	12	86.4	1.04
Ge g/t	89	75.0	0.92	177	2	68.0	0.62
Co ppm	89	182	2.76	672	7	125	1.51
Zone 7 – Fault Zone Splay							
Pb %	94	0.008	1.88	0.05	1	0.008	1.67
Co ppm	94	98	2.36	394	3	77	1.32
Zone 8 – Massive sulphide in Série Récurrenté Footwall							
Pb %	70	0.21	3.70	0.042	7	0.01	1.57
As %	70	0.58	1.95	2.44	2	0.46	1.49

Attribute	Before Capping			After Capping			
	Number of Composites	Mean	CV	Cap Value	Number of Composites Capped	Mean	CV
Ag g/t	70	135.5	1.69	698	1	121.9	1.42
Co ppm	70	259	3.14	552	3	130	1.20
Zone 9 – Low-grade zone in Fault Zone							
Cu %	37	056	1.37	1.81	2	0.50	0.95
Pb %	37	0.21	2.85	0.7	5	0.11	1.91
Zn %	37	7.12	1.32	17.5	2	5.95	0.95
As %	34	0.26	1.95	0.38	3	0.14	0.92
Zone 10 – Southern portion of Southern Zinc							
Pb %	55	0.05	1.88	0.07	9	0.03	0.97
Zn %	55	1.57	1.57	6.2	2	1.39	1.26
S %	55	8.51	1.24	19.7	5	6.67	0.81
Cd ppm	55	101	155	399	4	89	1.23

14.5 Geostatistical Analysis

14.5.1 Variograms

The 2 m composite data were examined using variograms that were calculated and modelled using Snowden Supervisor software. Most attributes were transformed to normal scores distributions and the spherical variogram models were back-transformed to normal statistical space for use in the grade interpolation process.

Variograms were calculated on the 2 m composite data and modelled within the plane of mineralisation with the minor direction being across strike. Rotations were aligned within each zone for all the attributes estimated. Normalised variograms were calculated, so that the sum of the variance (total sill value) is equal to one.

Variograms were modelled with either one, two or three spherical structures. The nugget effect was estimated by extrapolation of the first two experimental variogram points (calculated at the same lag as the composite length) to the Y axis.

For the Fault Zone and Southern Zinc, a plunge of 70° to the south-west within the plane of mineralisation was modelled. A plunge of 50° to the west was modelled for the Série Récurrenté and a vertical plunge was modelled for the Big Zinc grade continuity. Although the limits of the Big Zinc plunge steeply to the south-west, this trend was not evident in the grade continuity analysis. The plunge directions of the major zones were maintained for the minor zones, with the exception of the Série Récurrenté Footwall that has a plunge of 40° to the west. The directions of continuity were kept the same for all attributes within their respective zones.

There were insufficient data to calculate robust variograms for the Fault Zone Splay (Zone 7) and a variogram with a nugget effect of 0.03, a sill of 0.97 and a range of 40 m strike, 40 m dip and 10 m across strike was applied. The same variogram was used for the low-grade copper zone in the Fault Zone (Zone 9) as for the main mineralised portion (Zone 1). The Série Récurrenté Footwall zones (Zone 5 and Zone 8) were combined for variography purposes.

For the Big Zinc and Série Récurrenté 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. Fault Zone variograms tend to be more erratic with less well developed structure. The variograms for the smaller zones (Zone 3 and 5 to 10) are less robust there being fewer composites in these zones.

For all zones and attributes, the variogram ranges are in excess of the general drillhole spacing.

The variogram model parameters are shown in Table 14.7, after the variance has been back transformed from normal scores, and examples of normal scores variograms are shown in Figure 14.6 for Zone 2.

Figure 14.6 Zone 2 (Big Zinc) Zinc Variograms

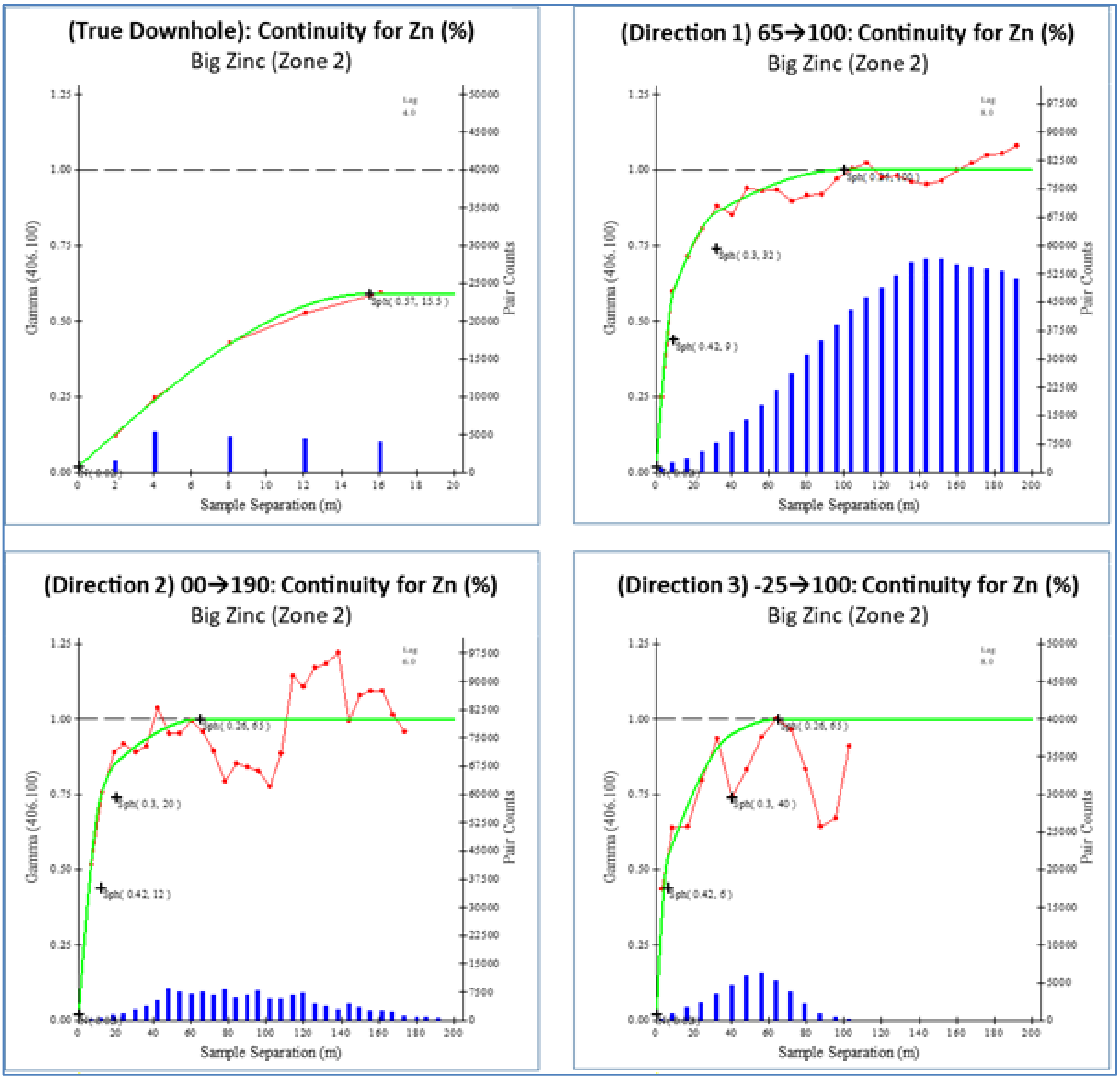


Figure by MSA, 2018.

Table 14.7 Variogram Parameters – Kipushi

Attribute	Transform	Rotation Angle			Nugget Effect (C0)	Range of First Structure (R1)			Sill 1 (C1)	Range of Second Structure (R2)			Sill 2 (C2)	Range of Third Structure (R3)			Sill 3 (C3)
		1	2	3		1	2	3		1	2	3		1	2	3	
Zone 1 and 8 – Fault Zone																	
Cu %	NS	-65	65	70	0.06	25	25	15	0.30	35	28	15	0.64	–	–	–	–
Pb %	NS	-65	65	70	0.03	15	15	9	0.30	60	60	19	0.67	–	–	–	–
Zn %	NS	-65	65	70	0.02	10	35	4	0.38	60	60	15	0.60	–	–	–	–
S %	NS	-65	65	70	0.01	27	36	5	0.33	30	36	11	0.66	–	–	–	–
As %	NS	-65	65	70	0.01	10	10	9	0.44	35	35	9	0.55	–	–	–	–
Ag g/t	NS	-65	65	70	0.14	7	7	4	0.43	28	18	15	0.43	–	–	–	–
Ge g/t	NS	-65	65	70	0.02	50	50	3	0.28	90	80	28	0.70	–	–	–	–
Co ppm	NS	-65	65	70	0.03	40	40	6	0.27	70	40	14	0.70	–	–	–	–
Cd ppm	NS	-65	65	70	0.02	11	11	6	0.48	90	90	15	0.50	–	–	–	–
Density	None	-65	65	70	0.04	15	15	6	0.40	45	45	9	0.56	–	–	–	–
Iron %	NS	-65	65	70	0.02	20	20	5	0.30	37	37	20	0.68	–	–	–	–
Zone 2 – Big Zinc																	
Cu %	NS	100	115	90	0.02	12	5	5	0.51	50	40	40	0.09	135	120	115	0.38
Pb %	NS	100	115	90	0.02	10	6	5	0.30	58	38	55	0.11	200	45	70	0.57
Zn %	None	100	115	90	0.02	9	12	6	0.43	32	20	40	0.30	100	65	65	0.26
S %	None	100	115	90	0.02	11	14	5	0.37	45	37	35	0.23	70	37	78	0.38
As %	NS	100	115	90	0.03	8	4	6	0.40	32	32	11	0.29	90	55	50	0.28
Ag g/t	NS	100	115	90	0.04	30	20	20	0.47	50	50	50	0.49	–	–	–	–
Ge g/t	NS	100	115	90	0.02	12	12	12	0.47	50	55	30	0.51	–	–	–	–
Co ppm	NS	100	115	90	0.04	12	7	5	0.28	64	33	10	0.32	64	64	42	0.36
Cd ppm	None	100	115	90	0.02	15	12	12	0.55	50	40	43	0.43	–	–	–	–
Density	None	100	115	90	0.04	18	18	10	0.48	65	47	47	0.48	–	–	–	–
Iron %	NS	100	115	90	0.04	10	10	10	0.47	105	90	50	0.49	–	–	–	–
Zone 3 – Southern Zinc																	
Cu %	NS	-65	65	70	0.08	38	40	25	0.92	–	–	–	–	–	–	–	–
Pb %	NS	-65	65	70	0.04	25	16	12	0.96	-	–	–	–	-	–	–	–
Zn %	None	-65	65	70	0.02	6	6	14	0.48	50	32	18	0.5	–	–	–	–
S %	None	-65	65	70	0.14	9	9	6	0.41	45	28	13	0.45	–	–	–	–
As %	NS	-65	65	70	0.06	50	50	20	0.94	–	–	–	–	–	–	–	–
Ag g/t	NS	-65	65	70	0.17	34	40	15	0.83	–	–	–	–	–	–	–	–
Ge g/t	NS	-65	65	70	0.05	10	10	10	0.49	52	52	15	0.46	–	–	–	–
Co ppm	NS	-65	65	70	0.17	33	32	15	0.83	–	–	–	–	–	–	–	–
Cd ppm	NS	-65	65	70	0.02	9	20	15	0.45	80	50	20	0.53	–	–	–	–
Density	None	-65	65	70	0.11	10	10	6	0.45	38	30	15	0.44	–	–	–	–
Iron %	None	-65	65	70	0.17	35	65	10	0.25	105	65	20	0.58	–	–	–	–
Zone 4 – Série Récurrenté																	
Cu %	NS	-170	95	-50	0.08	15	12	13	0.42	150	30	30	0.16	150	150	30	0.34
Pb %	NS	-170	95	-50	0.05	16	12	4	0.16	35	25	29	0.11	200	96	80	0.68
Zn %	NS	-170	95	-50	0.05	40	40	25	0.30	155	120	55	0.65	–	–	–	–
S %	NS	-170	95	-50	0.08	135	100	30	0.92	–	–	–	–	–	–	–	–
As %	NS	-170	95	-50	0.04	8	10	5	0.22	50	12	50	0.31	140	125	80	0.43
Ag g/t	NS	-170	95	-50	0.16	15	15	7	0.39	125	125	30	0.28	125	125	52	0.17
Ge g/t	NS	-170	95	-50	0.08	35	48	30	0.45	80	63	40	0.47	–	–	–	–
Co ppm	NS	-170	95	-50	0.34	30	22	7	0.35	130	110	60	0.65	–	–	–	–
Cd ppm	NS	-170	95	-50	0.02	25	20	40	0.33	130	110	60	0.65	–	–	–	–
Density	NS	-170	95	-50	0.02	90	75	70	0.98	–	–	–	–	–	–	–	–
Iron %	NS	-170	95	-50	0.07	130	95	30	0.93	–	–	–	–	–	–	–	–
Zone 5 and 8 – Série Récurrenté Footwall																	
Cu %	NS	-160	85	-40	0.72	50	30	9	0.28	–	–	–	–	–	–	–	–
Pb %	NS	-160	85	-40	0.10	45	30	10	0.90	–	–	–	–	–	–	–	–

Attribute	Transform	Rotation Angle			Nugget Effect (C0)	Range of First Structure (R1)			Sill 1 (C1)	Range of Second Structure (R2)			Sill 2 (C2)	Range of Third Structure (R3)			Sill 3 (C3)
		1	2	3		1	2	3		1	2	3		1	2	3	
Zn %	NS	-160	85	-40	0.13	50	40	9	0.87	–	–	–	–	–	–	–	–
S %	NS	-160	85	-40	0.48	50	45	4	0.52	–	–	–	–	–	–	–	–
As %	NS	-160	85	-40	0.55	25	25	6	0.45	–	–	–	–	–	–	–	–
Ag g/t	NS	-160	85	-40	0.81	30	30	4	0.19	–	–	–	–	–	–	–	–
Ge g/t	NS	-160	85	-40	0.04	50	45	7	0.96	–	–	–	–	–	–	–	–
Co ppm	NS	-160	85	-40	0.60	60	35	6	0.40	–	–	–	–	–	–	–	–
Cd ppm	NS	-160	85	-40	0.19	65	35	11	0.81	–	–	–	–	–	–	–	–
Density	NS	-160	85	-40	0.03	40	35	5	0.97	–	–	–	–	–	–	–	–
Iron %	NS	-160	85	-40	0.64	38	25	3	0.36	–	–	–	–	–	–	–	–
Zone 6 – High-grade Copper Zone Within Big Zinc																	
Cu %	NS	130	95	90	0.02	50	50	6	0.98	–	–	–	–	–	–	–	–
Pb %	NS	130	95	90	0.04	38	38	19	0.96	–	–	–	–	–	–	–	–
Zn %	NS	130	95	90	0.02	44	44	30	0.98	–	–	–	–	–	–	–	–
S %	NS	130	95	90	0.02	40	40	28	0.98	–	–	–	–	–	–	–	–
As %	NS	130	95	90	0.03	40	40	8	0.97	–	–	–	–	–	–	–	–
Ag g/t	NS	130	95	90	0.14	50	50	9	0.86	–	–	–	–	–	–	–	–
Ge g/t	NS	130	95	90	0.07	40	40	8	0.93	–	–	–	–	–	–	–	–
Co ppm	NS	130	95	90	0.07	40	40	8	0.93	–	–	–	–	–	–	–	–
Cd ppm	NS	130	95	90	0.05	36	36	30	0.95	–	–	–	–	–	–	–	–
Density	None	130	95	90	0.04	40	40	30	0.96	–	–	–	–	–	–	–	–
Iron %	None	130	95	90	0.02	43	43	20	0.98	–	–	–	–	–	–	–	–
Zone 7 – Fault Zone Splay																	
Cu %	None	90	90	90	0.03	40	40	10	0.97	–	–	–	–	–	–	–	–
Pb %	None	90	90	90	0.03	40	40	10	0.97	–	–	–	–	–	–	–	–
Zn %	None	90	90	90	0.03	40	40	10	0.97	–	–	–	–	–	–	–	–
S %	None	90	90	90	0.03	40	40	10	0.97	–	–	–	–	–	–	–	–
As %	None	90	90	90	0.03	40	40	10	0.97	–	–	–	–	–	–	–	–
Ag g/t	None	90	90	90	0.03	40	40	10	0.97	–	–	–	–	–	–	–	–
Ge g/t	None	90	90	90	0.03	40	40	10	0.97	–	–	–	–	–	–	–	–
Co ppm	None	90	90	90	0.03	40	40	10	0.97	–	–	–	–	–	–	–	–
Cd ppm	None	90	90	90	0.03	40	40	10	0.97	–	–	–	–	–	–	–	–
Density	None	90	90	90	0.03	40	40	10	0.97	–	–	–	–	–	–	–	–
Iron %	None	90	90	90	0.03	40	40	10	0.97	–	–	–	–	–	–	–	–
Zone 10 – Southern portion of Southern Zinc																	
Cu %	NS	-35	60	70	0.08	38	40	25	0.92	–	–	–	–	–	–	–	–
Pb %	NS	-35	60	70	0.04	25	16	12	0.96	–	–	–	–	–	–	–	–
Zn %	NS	-35	60	70	0.02	6	6	14	0.48	50	32	18	0.50	–	–	–	–
S %	NS	-35	60	70	0.14	9	9	6	0.45	45	28	13	0.45	–	–	–	–
As %	NS	-35	60	70	0.06	50	50	20	0.94	–	–	–	–	–	–	–	–
Ag g/t	NS	-35	60	70	0.17	34	40	15	0.83	–	–	–	–	–	–	–	–
Ge g/t	NS	-35	60	70	0.05	10	10	10	0.49	52	52	15	0.46	–	–	–	–
Co ppm	NS	-35	60	70	0.17	33	32	15	0.83	–	–	–	–	–	–	–	–
Cd ppm	NS	-35	60	70	0.02	9	20	15	0.45	80	50	20	0.53	–	–	–	–
Density	NS	-35	60	70	0.11	10	10	6	0.45	38	30	15	0.44	–	–	–	–
Iron %	NS	-35	60	70	0.17	35	65	10	0.25	105	65	20	0.58	–	–	–	–

All variograms are rotated on the Datamine Z-X-Z rotation logic.

14.5.2 Indicator Variograms

The mineralisation at Kipushi, in particular the Big Zinc, consists of extensive massive sulphide zones with internal 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. The Indicator approach was only necessary for the Fault Zone, Série Récurrenté, Big Zinc and Southern Zinc.

Indicator variograms were calculated using the 2 m sample composites and modelled at a threshold of 5% Zn for the zinc rich zones (Zone 2 and 3) and 0.5% Cu for the copper-rich zones (Zone 1 and Zone 4). The indicator variograms were modelled in three directions. The variogram models for Zone 2 and Zone 4 are robust and are informed by a reasonable number of experimental data, although the indictor variograms for Zone 1 and Zone 3 are poorly structured. The indicator variogram parameters are shown in Table 14.8.

Figure 14.7 Zone 2 - 5% Zinc Indicator Variograms

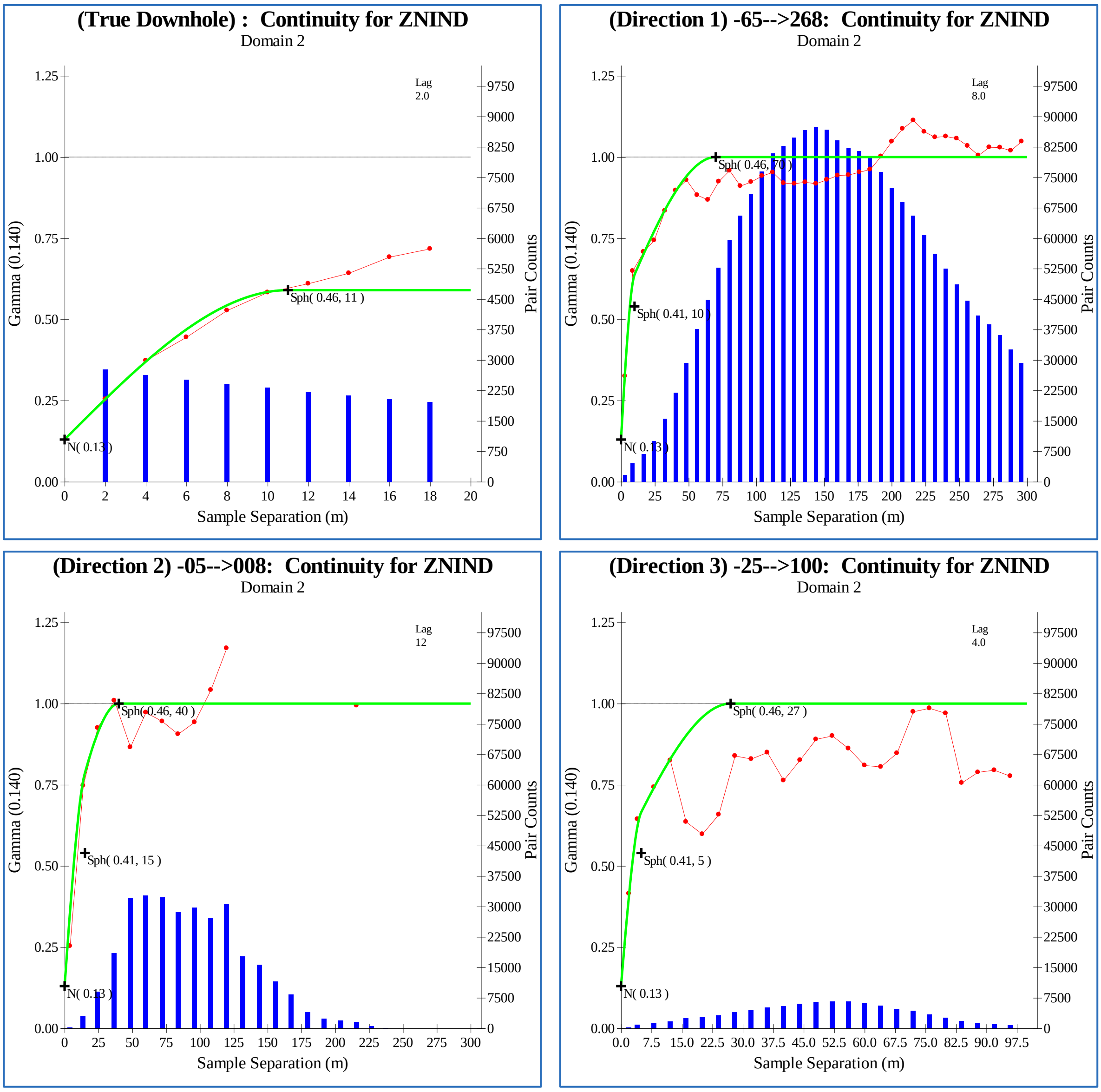


Figure by MSA, 2018

Figure 14.8 Zone 3 - 5% Zinc Indicator Variograms

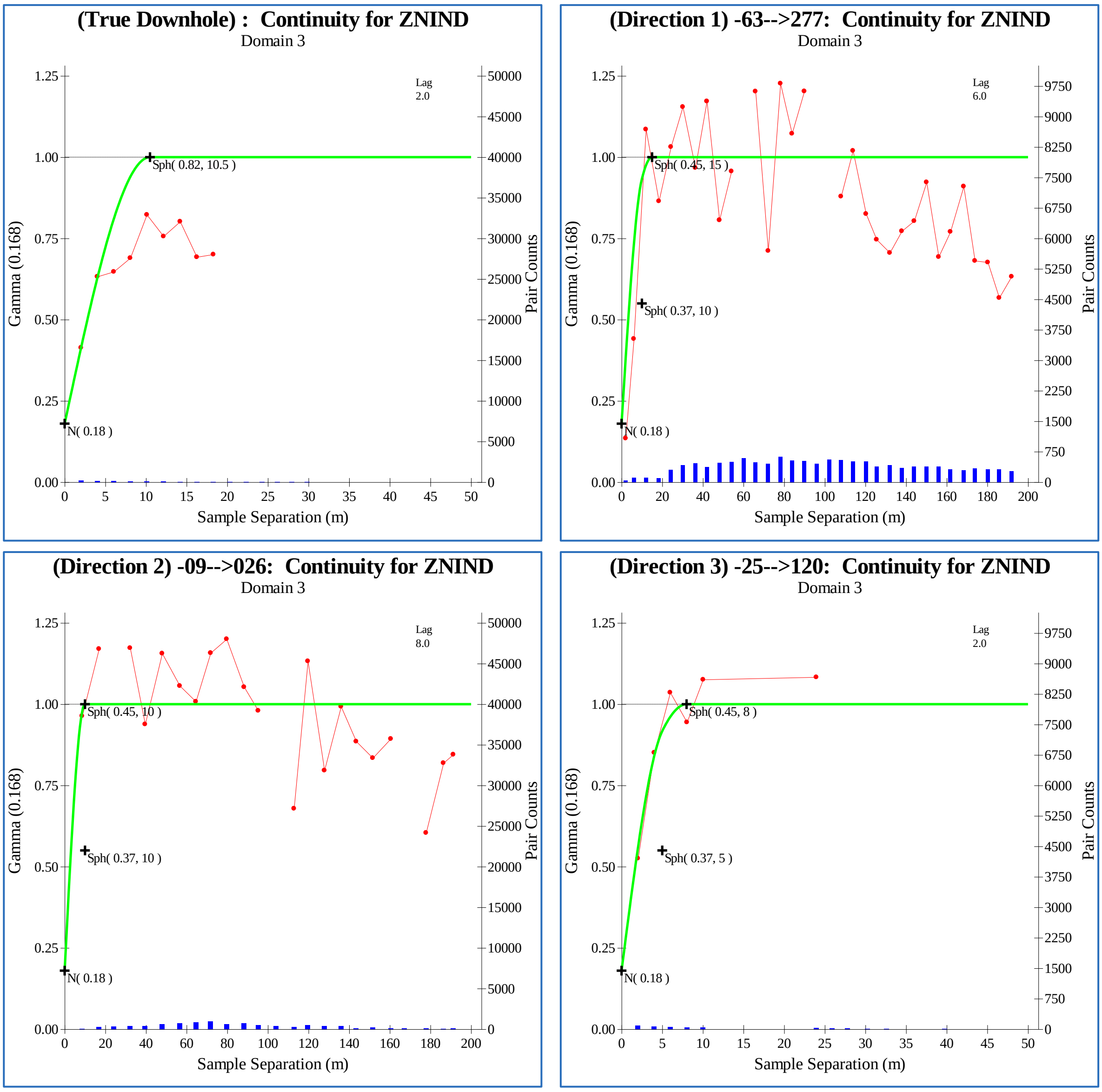


Figure by MSA, 2018.

Figure 14.9 Zone 4 - 0.5% Copper Indicator Variograms

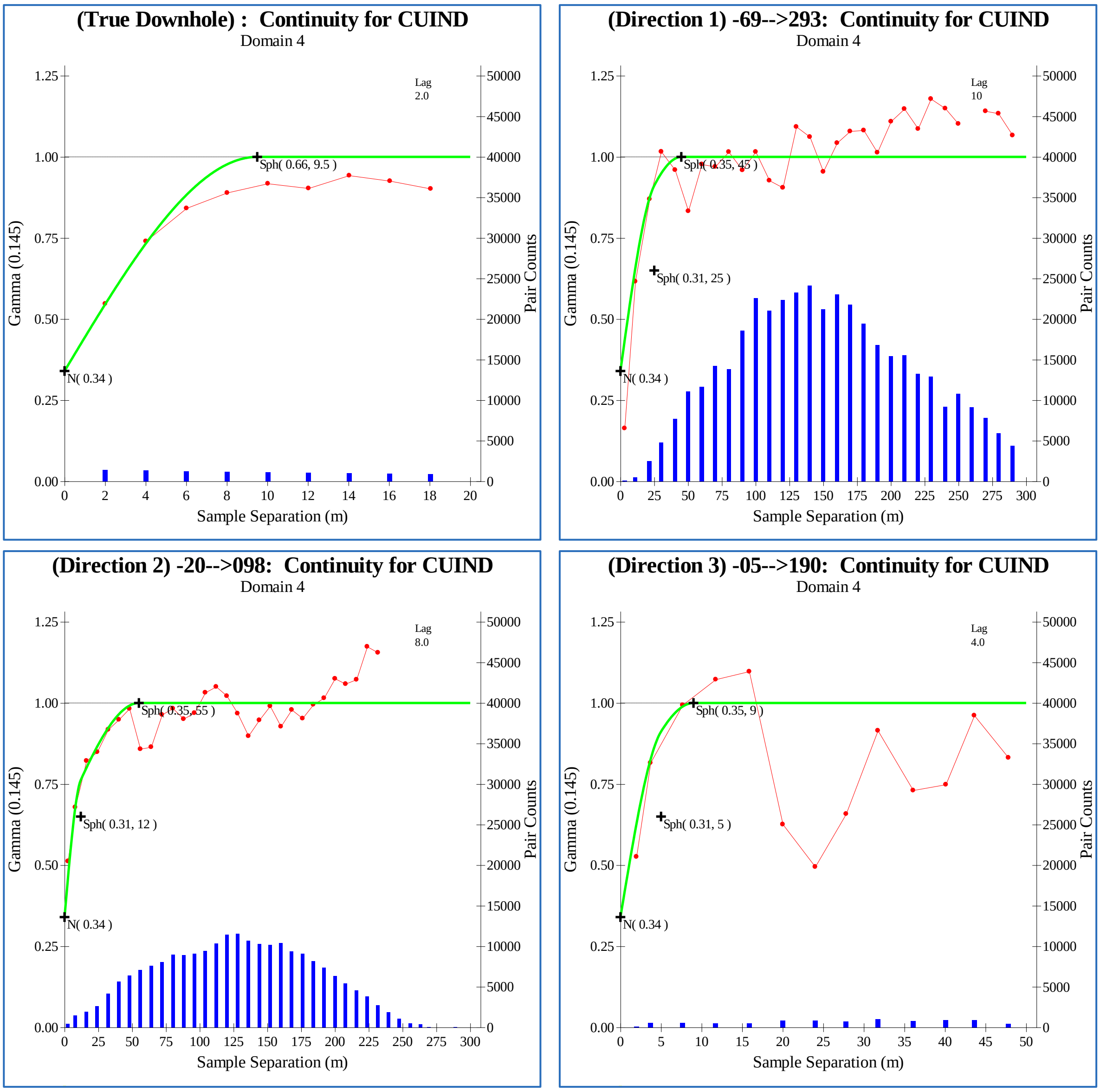


Figure by MSA, 2018.

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	115	115	-75	Z	X	Z	0.23	25	25	16	0.40	45	32	17	0.37
Big Zinc																
Zinc Indicator (0.5%)	None	100	115	-85	Z	X	Z	0.13	10	15	5	0.41	70	40	27	0.46
Southern Zinc																
Zn Indicator (5%)	None	120	115	-80	Z	X	Z	0.18	10	10	5	0.37	15	10	8	0.45
Série Récurrenté																
Cu Indicator (0.5%)	None	-170	95	-70	Z	X	Z	0.34	25	12	5	0.31	45	55	9	0.35

14.6 Block Modelling

The wireframes were filled with cells with a dimension of 5 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 a minimum of 0.5 mX by 0.5 mY by 0.5 mZ in order to best fill the irregular shapes of the mineralised bodies.

The ten different zone wireframes were filled separately, and the blocks were coded with the respective zone code.

The block model volume was compared to the wireframe volume and differences of less than 0.5% were found between the two, indicating that the wireframes were appropriately filled with block model cells.

14.7 Estimation

14.7.1 Indicator Estimation

In order to retain the high-grades in the massive zones and the low-grades in the isolated internal low-grade zones without smoothing the grades between them, an indicator approach was used to discriminate between them. The probability of a model cell being above or below a 0.5% Cu or 5% Zn threshold for the copper-rich and zinc-rich domains respectively was estimated using the 2 m composite data transformed to indicators, with "1" being above the threshold value and "0" being below. Ordinary kriging of the indicators into parent cells using the indicator variograms (Section 14.5.2) was carried out. The parameters used for the indicator estimation are shown in Table 14.9. These were aligned with the direction and distance of continuity as implied by the indicator variograms. Should an estimate not be achieved by selecting sufficient composites in the first search, the search was expanded until four composites were selected.

The Indicator approach was only necessary for the Fault Zone, Série Récurrenté, Big Zinc and Southern Zinc.

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

14.7.2 Grade Estimation

Each of the elements and density were estimated using ordinary kriging by estimating into parent cells.

The indicator estimates were carried out on the major element for each zone (copper for Zones 1 and 4 and zinc for Zones 2 and 3) and those closely related to them so that the indicator approach was applied to the following attributes:

- Zones 1 and 4 – copper, sulphur, iron and density.
- Zones 2 and 3 – zinc, cadmium, sulphur, iron and density.

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

The other attributes and zones were estimated using orginary kriging without indicators.

The search parameters used are shown in Table 14.10. A different search distance was allowed for each element, as the different elements tend to behave independently of each other. This is with the exception of cadmium and zinc, which are closely related, and the search parameter for zinc was applied to cadmium to ensure the relationship between these elements was preserved in the estimate.

The search parameters are based on the variogram ranges and anisotropy. The first search distance being the same as the total variogram range and the second search being 1.5 times the variogram range. A third search that sources a minimum of five and maximum of 10 samples was used. This is a greatly expanded search designed to achieve estimates approaching the local mean. A maximum of four composites from a single drillhole were allowed to estimate a cell in order to ensure that each cell was estimated using more than one drillhole. Any cells that were not estimated were assigned the domain average values.

The lead, arsenic, silver, germanium and cobalt distributions are characterised by small numbers of high-grade values within dominantly low-grade populations. The high-grades tend to occur in clusters. In order to retain the high-grade values locally, without smearing of the values throughout their respective estimation domains, a restricted omnidirectional search of 7 m was applied on the data during interpolation without capping applied. This allows the high-grades to influence only the block in which they occur and the immediately surrounding blocks. The estimates using the uncapped data replaced the estimates using the capped data. This technique honours areas of higher grade with short continuity and does not allow the higher-grades to influence areas that dominantly contain low or background level grade.

No arsenic, silver, germanium and cobalt assays were available for Zone 10 and the mean capped grades for Zone 1 were applied to it.

14.7.2.1 Boundary Conditions

Each domain was estimated only using the drillhole data within it (hard boundaries).

Table 14.10 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 and Zone 9)																	
Cu %	-65	65	70	Z	X	Z	35	28	15	6	12	1.5	6	12	100	5	10
Pb g/t	-65	65	70	Z	X	Z	60	60	19	6	12	1.5	6	12	100	5	10
Zn %	-65	65	70	Z	X	Z	60	60	15	6	12	1.5	6	12	100	5	10
S %	-65	65	70	Z	X	Z	30	36	11	6	12	1.5	6	12	100	5	10
As %	-65	65	70	Z	X	Z	35	35	9	6	12	1.5	6	12	100	5	10
Ag g/t	-65	65	70	Z	X	Z	28	18	15	6	12	1.5	6	12	100	5	10
Ge g/t	-65	65	70	Z	X	Z	90	80	28	6	12	1.5	6	12	100	5	10
Co ppm	-65	65	70	Z	X	Z	70	40	14	6	12	1.5	6	12	100	5	10
Cd ppm	-65	65	70	Z	X	Z	60	60	15	6	12	1.5	6	12	100	5	10
Density	-65	65	70	Z	X	Z	45	45	9	6	12	1.5	6	12	100	5	10
Fe %	-65	65	70	Z	X	Z	37	37	20	6	12	1.5	6	12	100	5	10
Big Zinc (Zone 2)																	
Cu %	100	115	90	Z	X	Z	135	120	115	6	12	1.5	6	12	100	5	10
Pb g/t	100	115	90	Z	X	Z	200	45	70	6	12	1.5	6	12	100	5	10
Zn %	100	115	90	Z	X	Z	100	65	35	6	12	1.5	6	12	100	5	10
S %	100	115	90	Z	X	Z	70	37	78	6	12	1.5	6	12	100	5	10

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.
As %	100	115	90	Z	X	Z	90	55	50	6	12	1.5	6	12	100	5	10
Ag g/t	100	115	90	Z	X	Z	50	50	50	6	12	1.5	6	12	100	5	10
Ge g/t	100	115	90	Z	X	Z	50	55	30	6	12	1.5	6	12	100	5	10
Co ppm	100	115	90	Z	X	Z	64	64	32	6	12	1.5	6	12	100	5	10
Cd ppm	100	115	90	Z	X	Z	100	65	65	6	12	1.5	6	12	100	5	10
Density	100	115	90	Z	X	Z	65	47	47	6	12	1.5	6	12	100	5	10
Fe %	100	115	90	Z	X	Z	105	90	50	6	12	1.5	6	12	100	5	10
Southern Zinc (Zone 3)																	
Cu %	-60	65	70	Z	X	Z	38	40	25	6	12	1.5	6	12	100	5	10
Pb g/t	-60	65	70	Z	X	Z	25	16	12	6	12	1.5	6	12	100	5	10
Zn %	-60	65	70	Z	X	Z	50	32	18	6	12	1.5	6	12	100	5	10
S %	-60	65	70	Z	X	Z	45	28	13	6	12	1.5	6	12	100	5	10
As %	-60	65	70	Z	X	Z	50	50	20	6	12	1.5	6	12	100	5	10
Ag g/t	-60	65	70	Z	X	Z	34	40	15	6	12	1.5	6	12	100	5	10
Ge g/t	-60	65	70	Z	X	Z	52	52	15	6	12	1.5	6	12	100	5	10
Co ppm	-60	65	70	Z	X	Z	33	32	15	6	12	1.5	6	12	100	5	10
Cd ppm	-60	65	70	Z	X	Z	50	32	18	6	12	1.5	6	12	100	5	10
Density	-60	65	70	Z	X	Z	38	30	15	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.
Fe %	-60	65	70	Z	X	Z	105	65	20	6	12	1.5	6	12	100	5	10
Série Récurrente (Zone 4)																	
Cu %	-170	95	50	Z	X	Z	150	150	30	6	12	1.5	6	12	100	5	10
Pb g/t	-170	95	50	Z	X	Z	200	96	80	6	12	1.5	6	12	100	5	10
Zn %	-170	95	50	Z	X	Z	155	120	55	6	12	1.5	6	12	100	5	10
S %	-170	95	50	Z	X	Z	135	100	30	6	12	1.5	6	12	100	5	10
As %	-170	95	50	Z	X	Z	140	125	80	6	12	1.5	6	12	100	5	10
Ag g/t	-170	95	50	Z	X	Z	125	125	52	6	12	1.5	6	12	100	5	10
Ge g/t	-170	95	50	Z	X	Z	80	63	40	6	12	1.5	6	12	100	5	10
Co ppm	-170	95	50	Z	X	Z	58	25	17	6	12	1.5	6	12	100	5	10
Cd ppm	-170	95	50	Z	X	Z	155	120	55	6	12	1.5	6	12	100	5	10
Density	-170	95	50	Z	X	Z	90	75	40	6	12	1.5	6	12	100	5	10
Fe %	-170	95	50	Z	X	Z	180	70	39	6	12	1.5	6	12	100	5	10
High-grade Zone in Série Récurrente (Zone 5 and Zone 8)																	
Cu %	-160	85	-40	Z	X	Z	50	30	9	6	12	1.5	6	12	100	5	10
Pb g/t	-160	85	-40	Z	X	Z	45	30	10	6	12	1.5	6	12	100	5	10
Zn %	-160	85	-40	Z	X	Z	50	40	9	6	12	1.5	6	12	100	5	10
S %	-160	85	-40	Z	X	Z	50	45	4	6	12	1.5	6	12	100	5	10

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.
As %	-160	85	-40	Z	X	Z	25	25	6	6	12	1.5	6	12	100	5	10
Ag g/t	-160	85	-40	Z	X	Z	35	30	4	6	12	1.5	6	12	100	5	10
Ge g/t	-160	85	-40	Z	X	Z	50	45	7	6	12	1.5	6	12	100	5	10
Co ppm	-160	85	-40	Z	X	Z	60	35	6	6	12	1.5	6	12	100	5	10
Cd ppm	-160	85	-40	Z	X	Z	50	40	9	6	12	1.5	6	12	100	5	10
Density	-160	85	-40	Z	X	Z	40	35	5	6	12	1.5	6	12	100	5	10
Fe %	-160	85	-40	Z	X	Z	38	25	3	6	12	1.5	6	12	100	5	10
Copper –rich zone in Big Zinc (Zone 6)																	
Cu %	130	95	90	Z	X	Z	50	50	6	6	12	1.5	6	12	100	5	10
Pb g/t	130	95	90	Z	X	Z	38	38	19	6	12	1.5	6	12	100	5	10
Zn %	130	95	90	Z	X	Z	44	44	30	6	12	1.5	6	12	100	5	10
S %	130	95	90	Z	X	Z	40	40	28	6	12	1.5	6	12	100	5	10
As %	130	95	90	Z	X	Z	40	40	8	6	12	1.5	6	12	100	5	10
Ag g/t	130	95	90	Z	X	Z	50	50	9	6	12	1.5	6	12	100	5	10
Ge g/t	130	95	90	Z	X	Z	40	40	8	6	12	1.5	6	12	100	5	10
Co ppm	130	95	90	Z	X	Z	65	65	28	6	12	1.5	6	12	100	5	10
Cd ppm	130	95	90	Z	X	Z	44	44	30	6	12	1.5	6	12	100	5	10
Density	130	95	90	Z	X	Z	40	40	30	6	12	1.5	6	12	100	5	10

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.
Fe %	130	95	90	Z	X	Z	43	43	20	6	12	1.5	6	12	100	5	10
Fault Splay Zone (Zone 7)																	
Cu %	90	90	90	Z	X	Z	40	40	10	6	12	1.5	6	12	100	5	10
Pb g/t	90	90	90	Z	X	Z	40	40	10	6	12	1.5	6	12	100	5	10
Zn %	90	90	90	Z	X	Z	40	40	10	6	12	1.5	6	12	100	5	10
S %	90	90	90	Z	X	Z	40	40	10	6	12	1.5	6	12	100	5	10
As %	90	90	90	Z	X	Z	40	40	10	6	12	1.5	6	12	100	5	10
Ag g/t	90	90	90	Z	X	Z	40	40	10	6	12	1.5	6	12	100	5	10
Ge g/t	90	90	90	Z	X	Z	40	40	10	6	12	1.5	6	12	100	5	10
Co ppm	90	90	90	Z	X	Z	40	40	10	6	12	1.5	6	12	100	5	10
Cd ppm	90	90	90	Z	X	Z	40	40	10	6	12	1.5	6	12	100	5	10
Density	90	90	90	Z	X	Z	40	40	10	6	12	1.5	6	12	100	5	10
Fe %	90	90	90	Z	X	Z	40	40	10	6	12	1.5	6	12	100	5	10
Copper-rich zone in Southern Zinc (Zone 10)																	
Cu %	-35	60	70	Z	X	Z	38	40	25	6	12	1.5	6	12	100	5	10
Pb g/t	-35	60	70	Z	X	Z	25	16	12	6	12	1.5	6	12	100	5	10
Zn %	-35	60	70	Z	X	Z	50	32	18	6	12	1.5	6	12	100	5	10
S %	-35	60	70	Z	X	Z	45	28	13	6	12	1.5	6	12	100	5	10

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.
As %	-35	60	70	Z	X	Z	50	50	20	6	12	1.5	6	12	100	5	10
Ag g/t	-35	60	70	Z	X	Z	34	40	15	6	12	1.5	6	12	100	5	10
Ge g/t	-35	60	70	Z	X	Z	52	52	15	6	12	1.5	6	12	100	5	10
Co ppm	-35	60	70	Z	X	Z	33	32	15	6	12	1.5	6	12	100	5	10
Cd ppm	-35	60	70	Z	X	Z	50	32	18	6	12	1.5	6	12	100	5	10
Density	-35	60	70	Z	X	Z	38	30	15	6	12	1.5	6	12	100	5	10
Fe %	-35	60	70	Z	X	Z	105	65	20	6	12	1.5	6	12	100	5	10

14.8 Validation of the Estimates

The models were validated by:

- Visual examination of the input data against the block model estimates,
- Sectional validation,
- Comparison of the input data statistics against the model statistics.

The block model was examined visually in sections to ensure that the drillhole grades were locally well represented by the model. It was found that the model validated reasonably well against the data. A section showing the block model and drillholes is shown in Figure 14.10.

Figure 14.10 Section through Big Zinc and Fault Zone Block Model and Drillhole Data Illustrating Correlation between Model and Data, shaded by Zinc (Left) and Copper (Right)

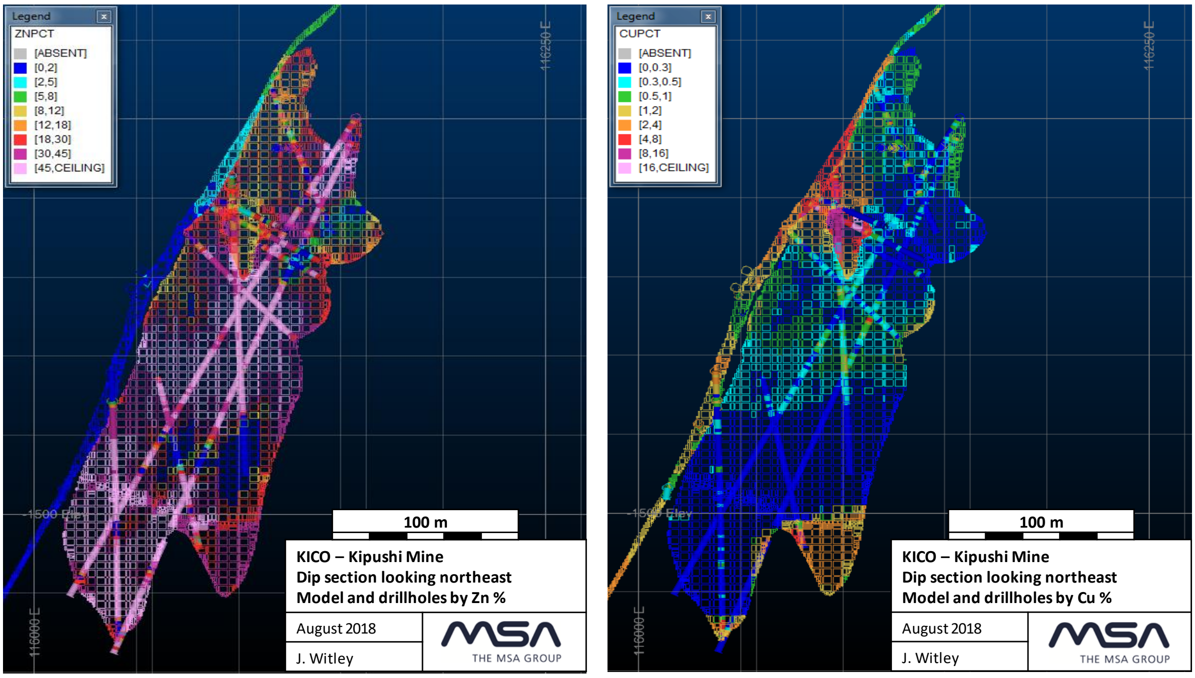


Figure by MSA, 2018.

Sectional validation plots were constructed for each major element and each zone. The sectional validation plots compare the average grades of the block model against the input data along a number of corridors in various directions through the deposit. Samples of the sectional validation plots are shown in Figure 14.11. These show that the estimates retain the local grade trends across the deposit.

Figure 14.11 Sectional Validation Plots

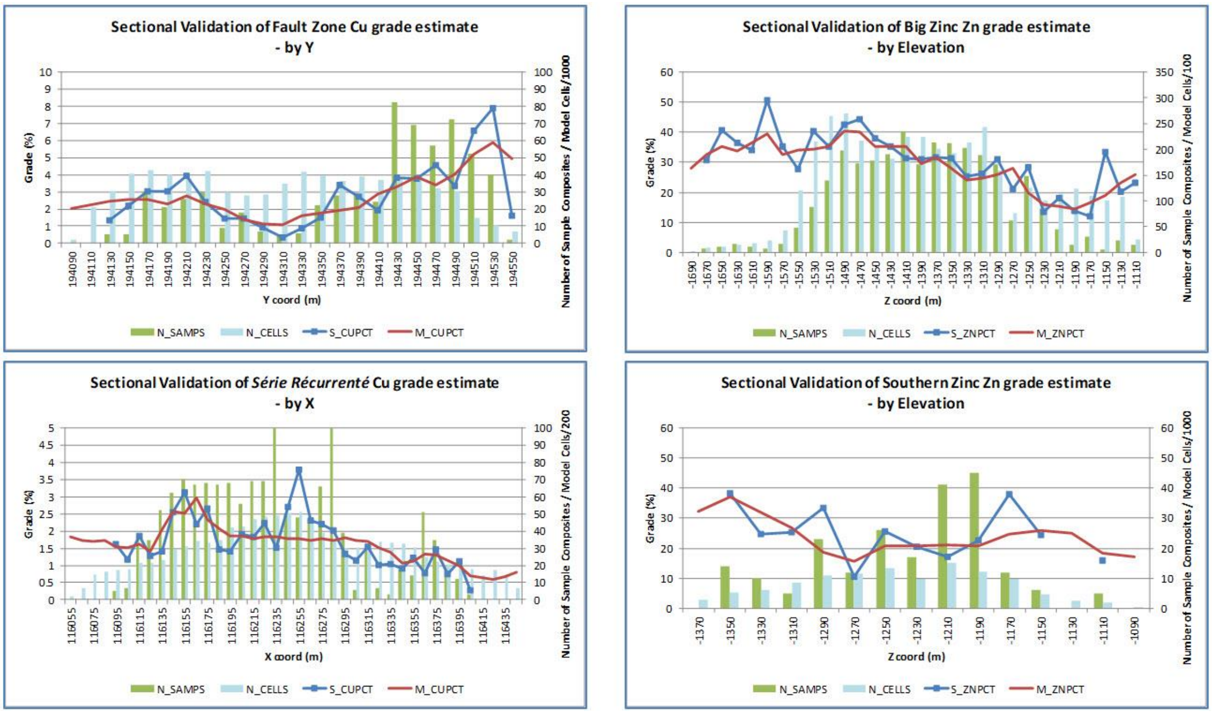


Figure by MSA, 2018.

As a further check, the declustered drillhole composite mean grades were compared with the model grade. The model and the data averages compare reasonably well for most variables. Those that did not compare within reasonable limits were examined further. No consistent biases were found, and the differences were all explained by the arrangement of the data relative to the volume of the model and are of no concern. For the elements that were estimated using the restricted uncapped search (lead, arsenic, silver, germanium and cobalt) higher discrepancies between the capped mean and model mean tended to occur. The more significant discrepancies between the capped mean and model mean are explained as follows.

- The Zone 1 germanium model grade is 62.3% higher than the capped mean and is 36% higher than the uncapped mean. Only the KICO drillholes were assayed for germanium and a large proportion of the model was outside of the KICO drilling area. The data on the fringes of the KICO drilling area, which are higher than the data mean, have been extrapolated to the south-west.
- The lead estimates for several zones are significantly higher than the capped mean but lower than the uncapped mean. This is a function of the restrictive search on the uncapped data.
- The germanium model grade for Zone 7 is 33.5% higher than the data mean. The data for this model is sparse and irregularly spaced and is the estimate is therefore very susceptible to the data arrangement. Relatively high-grades have been extrapolated into a large poorly informed area to the north. This is also the case for cadmium
- The zinc model grade for Zone 8 is 33.1% higher than the mean data grade. For

cadmium the model grade is 34.3% higher than the mean data grade. The lower-grade data tends to occur on the edges of the model and therefore have less influence than the higher-grade data that occur towards the centre of the model. The model was examined in detail visually and with sectional validation plots and no issues were found.

14.9 Mineral Resource Classification

Classification of the Kipushi Mineral Resource was based on confidence in the data, confidence in the geological model, grade continuity and variability and the frequency of the drilling data. The main considerations in the classification of the Kipushi Mineral Resource are as follows:

- The data were collected by KICO and Gécamines. The KICO data have been collected using current industry standard principles; however, the quality of the Gécamines data is less certain. KICO has endeavoured to verify the Gécamines data by a programme of re-sampling and twin drilling in the Big Zinc and portions of the Fault Zone which yielded reasonable comparisons.
- The Gécamines data are incomplete in several aspects; notably not all of the elements of interest were analysed and the sampling was selective in some of the drillholes. A rigorous validation exercise was completed that resulted in many of the Gécamines holes being rejected for use in the grade estimate.
- Areas of the Fault Zone, Série Récurrenté and the southern portion of the Southern Zinc are only informed by Gécamines drillholes. The Big Zinc has been well drilled by KICO as well as a portion of the Série Récurrenté and the Fault Zone.
- The geological framework of the Mineral Resource is well understood as are the controls to the mineralisation.
- The Mineral Resource has been densely drilled on sections spaced 15 m apart, although areas of the Série Récurrenté and down dip areas of the Fault Zone are less well drilled.
- Variogram ranges are well in excess of the drillhole spacing.
- The grade model validates reasonably well, although suffers from a lack of data for several elements notably silver, germanium and cobalt, as these assays were not available in the database constructed from the Gécamines data.
- Kipushi has an extensive mining history and the continuity of the mineralised bodies has been established through mining.

Given the aforementioned factors the Kipushi Mineral Resource was classified using the following criteria:

- One area of the Big Zinc and adjacent Fault Zone was classified as Measured. The spacing of the KICO drillholes in this area is less than 20 m and there is high confidence in the interpretation of the mineralised extents.

- Where informed predominantly by KICO drilling, and with a drillhole spacing of closer than 50 m, the Mineral Resource was classified as Indicated. This applies to the majority of the Big Zinc, the Fault Zone in the vicinity of the Big Zinc and Southern Zinc, the northern and central; portions of the Southern Zinc and an area of the Série Récurrenté. Consideration of the proximity to the areas of historical mining was made, as in general these will be of lower risk.
- For areas of the Mineral Resource predominantly informed by Gécamines drillholes, the Mineral Resource was classified as Inferred. This applies to the southern portion of the Southern Zinc and areas of the Fault Zone and Série Récurrenté.
- The Fault Zone Splay was classified as Inferred. This zone is informed by six KICO drillholes, many of which are drilled at a close angle to the plane of the mineralisation. Grades in this area are variable and the interpretation of the mineralised extents is tenuous.
- Extrapolation of the Big Zinc was limited to a maximum of 15 m, the complex shape of the deposit negated against greater extrapolation with any confidence. The Fault Zone and Série Récurrenté are highly continuous and the down dip extent was limited to 50 m from the drillhole intersections.

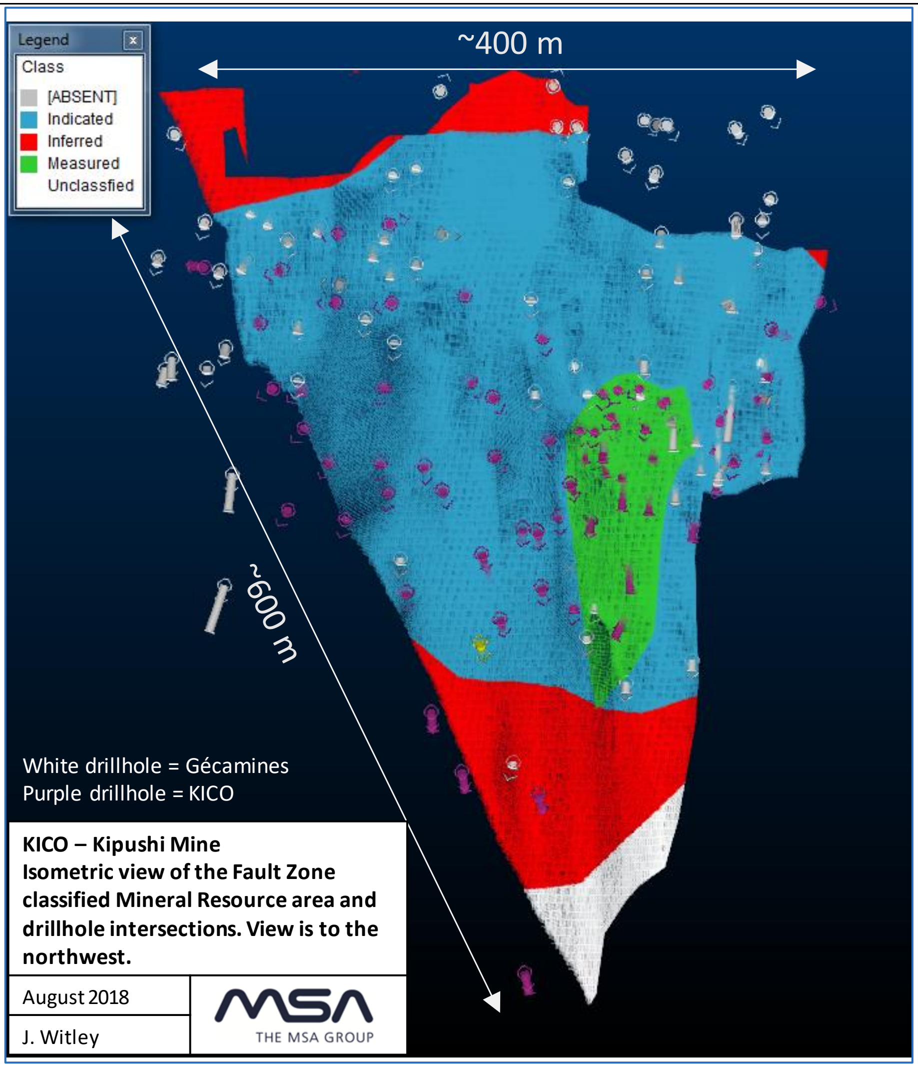
The classified areas for the Big Zinc, for the Fault Zone and for the Série Récurrenté are shown in Figure 14.12.

To the best of the Qualified Person's knowledge there is no environmental, permitting, legal, tax, socio-political, marketing or other relevant issues which may materially affect the Mineral Resource estimate as reported in the Kipushi 2019 Resource Update, aside from those mentioned in Section 4 of this report.

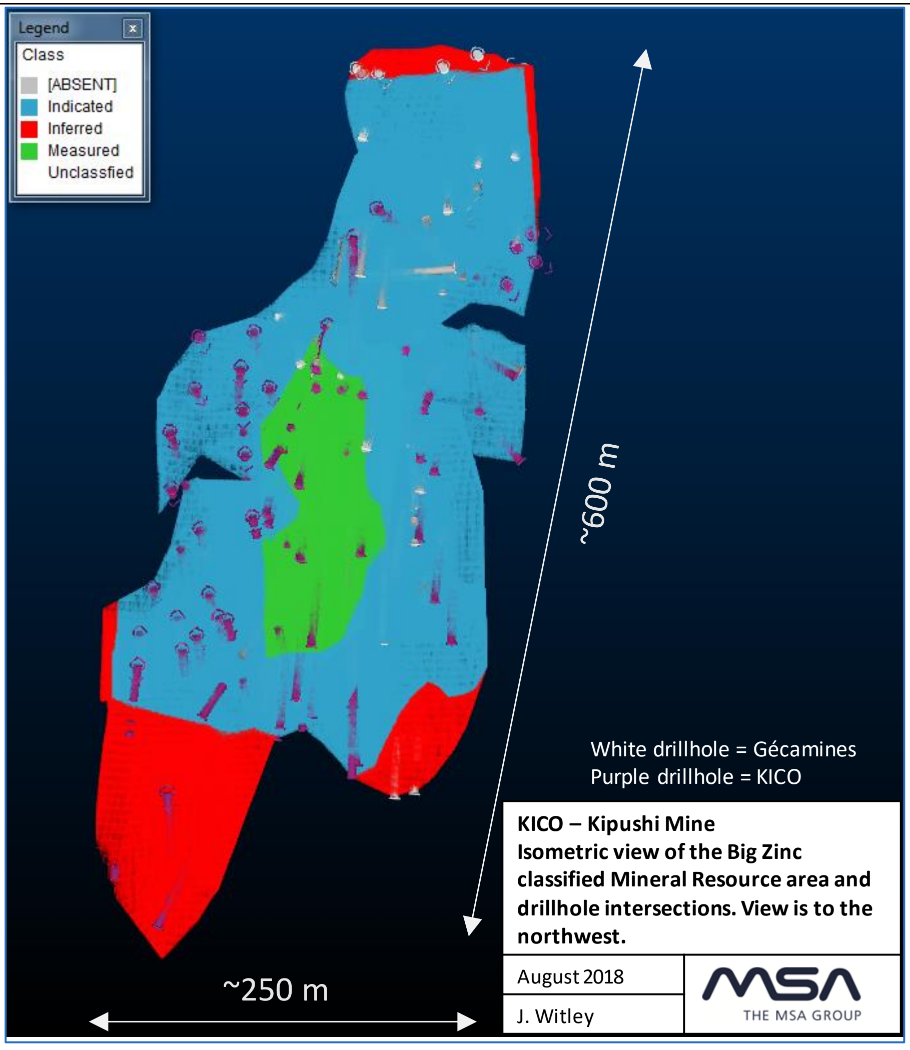
The Mineral Resources could be affected by further infill and exploration drilling, which may result in increases or decreases in subsequent Mineral Resource estimates. Inferred Mineral Resources are considered to be high risk estimates that may change significantly with additional data. It cannot be assumed that all or part of an Inferred Mineral Resource will necessarily be upgraded to an Indicated Mineral Resource as a result of continued exploration. The Mineral Resources may also be affected by subsequent assessments of mining, environmental, processing, permitting, taxation, socio-economic, and other factors.

Figure 14.12 Mineral Resource Classification

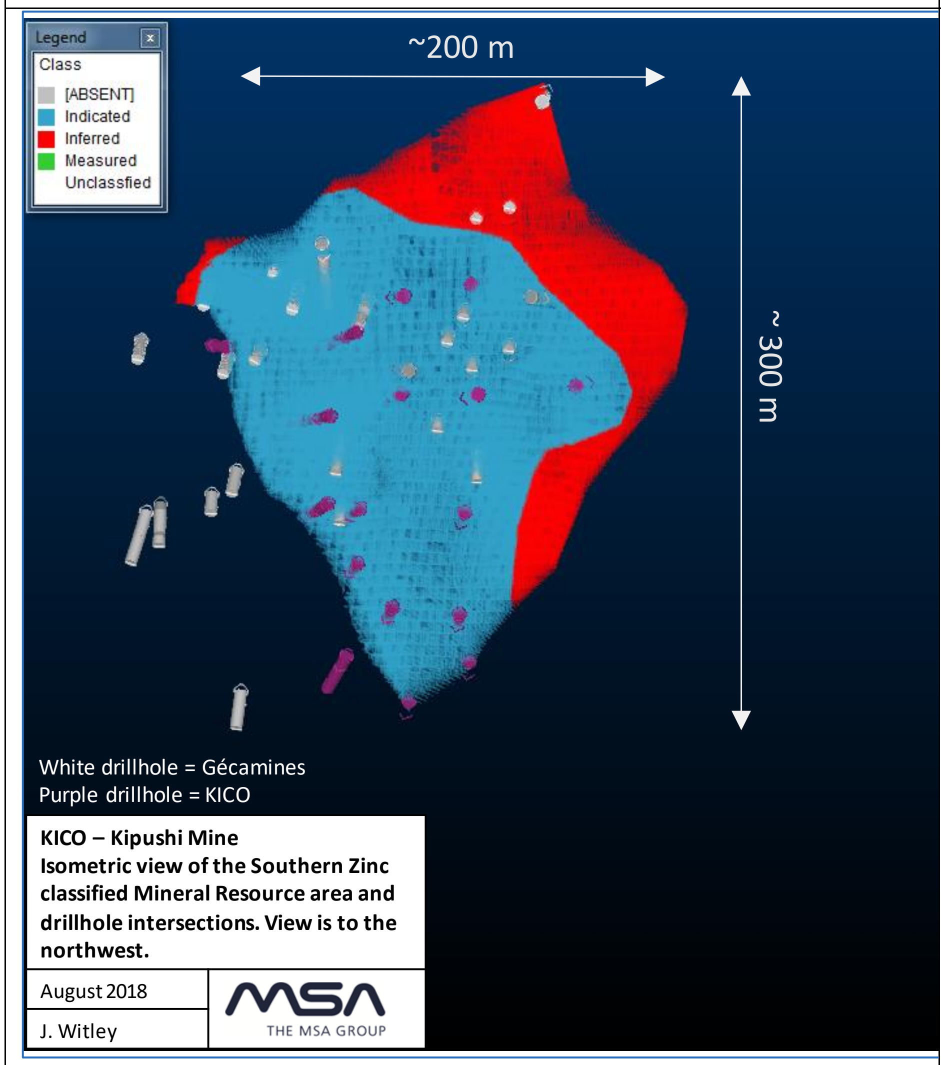
Fault Zone – Isometric View Looking Approximately North-west



Big Zinc – Isometric View Looking Approximately East



Southern Zinc – Isometric View Looking Approximately North-west



Série Récurrenté – Isometric View Looking Approximately South-east

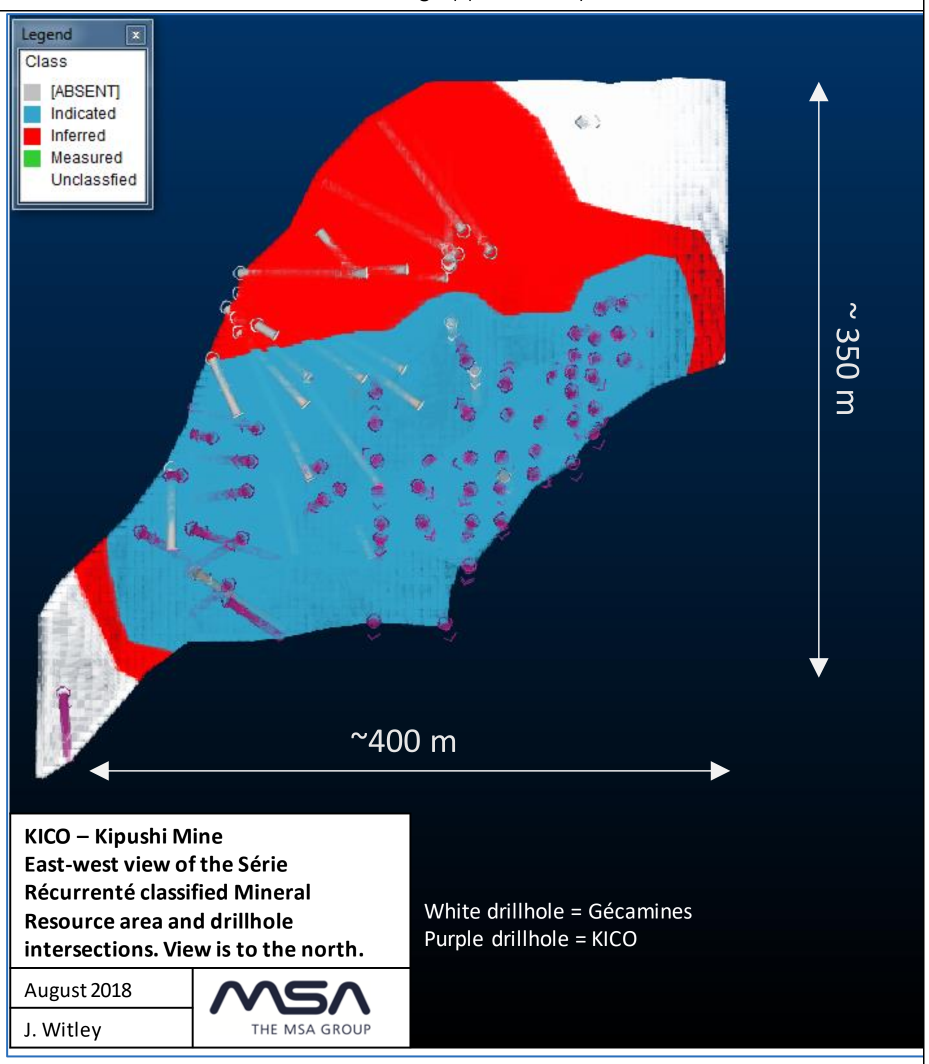


Figure by MSA, 2018. Only drillholes used for estimation shown. Only area in DRC shown.

14.10 Depletion of the Mineral Resource

The grade model includes areas that have previously been mined by Gécamines and an area to the south-west inside Zambia.

14.10.1 Mined out Areas

Mined out areas were supplied by KICO. These were simplified into cohesive areas, so that isolated remnants were not included in the Mineral Resource estimate, and then used for depletion of the model. In addition, the entire model above 1,150 mRL was removed, extensive mining having taken place in that region. There is potential for additional Mineral Resources to exist above 1,150 mRL but this will require investigation in terms of mineralisation remaining and reasonable prospects for eventual economic extraction of the remnant areas.

14.10.2 Zambia-DRC Border

The mineralisation at Kipushi straddles the DRC-Zambia border, however, the exact position of the border is uncertain at Kipushi, as there is currently 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 Tshwane, South Africa) to determine the position of the border as accurately as possible (Cochran, 2015).

Mr Cochran located the position of four of the original border beacons (probably from the early 1930's) and surveyed them using high precision GNSS post processing systems (on ITRF2008/WGS84). Together with information obtained by interviewing local inhabitants and from the Zambian Department of Survey and Lands in Lusaka, a pragmatic border line was interpreted (Figure 14.13). Mr Cochran is confident that the pragmatic border line best represents the most likely border line. The interpreted border line generally fits to the surveyed beacons to within +/-0.5 m and follows the general trend of the watershed in the area.

Figure 14.13 Google Earth Image Showing Position of DRC-Zambia Border

The border from Google Earth is shown in yellow and the pragmatic border line in green.
Source- Google Earth and Cochran, 2015.

The pragmatic border line was projected vertically to the Kipushi mineralisation models and all modelled mineralisation on the Zambian side of the border line was removed from the Mineral Resource estimate.

14.11 Mineral Resource Statement

The Mineral Resource was estimated using The Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Best Practice Guidelines and is reported in accordance with the 2014 CIM Definition Standards, which have been incorporated by reference into National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101). The Mineral Resource is classified into the Measured, Indicated and Inferred categories as shown in Table 14.11 for the predominantly zinc-rich bodies and in Table 14.12 for the predominantly copper-rich bodies.

The Measured 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 14.13 and Table 14.14 respectively and Table 14.15 and Table 14.16 for the copper-rich bodies.

For the zinc-rich zones the Mineral Resource is reported at a base case cut-off grade of 7.0% Zn, and the copper-rich zones at a base case cut-off grade of 1.5% Cu. Given the considerable revenue which will be obtained from the additional metals in each zone, MSA considers that mineralisation at these cut-off grades will satisfy reasonable prospects for economic extraction.

It should be noted that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability and the economic parameters used to assess the potential for economic extraction is not an attempt to estimate Mineral Reserves.

Table 14.11 Kipushi Zinc-Rich Mineral Resource at 7% Zn Cut-off Grade, 14 June 2018

Zone	Category	Tonnes (Millions)	Zn (%)	Cu (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
Big Zinc	Measured	3.65	39.87	0.65	0.35	18	18	56
	Indicated	7.25	34.36	0.62	1.29	19	12	53
	Inferred	0.98	35.32	1.18	0.09	8	15	62
Southern Zinc	Indicated	0.88	24.52	2.97	1.95	75	6	188
	Inferred	0.16	24.37	1.64	1.20	38	6	61
Total	**Measured**	**3.65**	**39.87**	**0.65**	**0.35**	**18**	**18**	**56**
	Indicated	**8.13**	**33.30**	**0.87**	**1.36**	**25**	**11**	**68**
	Measured and Indicated	**11.78**	**35.34**	**0.80**	**1.05**	**23**	**13**	**64**
	Inferred	**1.14**	**33.77**	**1.24**	**0.24**	**12**	**14**	**62**

		Contained Metal Quantities						
Zone	**Category**	**Tonnes (Millions)**	**Zn Pounds (Millions)**	**Cu Pounds (Millions)**	**Pb Pounds (Millions)**	**Ag Ounces (Millions)**	**Co Pounds (Millions)**	**Ge Ounces (Millions)**
Big Zinc	Measured	3.65	3,210.6	52.3	27.8	2.06	0.14	6.60
	Indicated	7.25	5,489.0	98.7	206.6	4.48	0.19	12.43
	Inferred	0.98	764.0	25.5	1.9	0.26	0.03	1.96
Southern Zinc	Indicated	0.88	476.5	57.6	37.8	2.11	0.01	5.34
	Inferred	0.16	86.7	5.8	4.3	0.20	0.00	0.32
Total	**Measured**	**3.65**	**3,210.6**	**52.3**	**27.8**	**2.06**	**0.14**	**6.60**
	Indicated	**8.13**	**5,965.5**	**156.4**	**244.4**	**6.59**	**0.20**	**17.77**
	Measured and Indicated	**11.78**	**9,176.0**	**208.6**	**272.2**	**8.65**	**0.34**	**24.36**
	Inferred	**1.14**	**850.7**	**31.3**	**6.2**	**0.46**	**0.04**	**2.28**

Note:

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

Table 14.12 Kipushi Copper-Rich Mineral Resource at 1.5% Cu Cut-off grade, 14 June 2018

Zone	Category	Tonnes (Millions)	Cu (%)	Zn (%)	Pb (%)	Ag (g/t)	Co (ppm)	Ge (g/t)
Fault Zone	Measured	0.14	2.74	1.52	0.04	16	77	21
	Indicated	1.22	4.11	3.32	0.09	21	96	30
	Inferred	0.20	3.11	2.58	0.07	18	43	23
Série Récurrenté	Indicated	0.93	4.14	2.43	0.02	23	50	4
	Inferred	0.03	1.81	0.06	0.00	8	52	0.3
Fault Zone Splay	Inferred	0.21	4.91	19.84	0.01	21	107	93
Total	**Measured**	**0.14**	**2.74**	**1.52**	**0.04**	**16**	**77**	**21**
	Indicated	**2.15**	**4.12**	**2.94**	**0.06**	**22**	**76**	**19**
	Measured and Indicated	**2.29**	**4.03**	**2.85**	**0.06**	**21**	**76**	**19**
	Inferred	**0.44**	**3.89**	**10.77**	**0.04**	**19**	**75**	**55**

		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	4.7	0.1	0.07	0.02	0.09
	Indicated	1.22	110.8	89.7	2.5	0.82	0.26	1.19
	Inferred	0.20	13.4	11.1	0.3	0.12	0.02	0.14
Série Récurrenté	Indicated	0.93	84.6	49.8	0.5	0.69	0.10	0.12
	Inferred	0.03	1.3	0.04	0.0	0.01	0.00	0.00
Fault Zone Splay	Inferred	0.21	23.2	93.7	0.1	0.14	0.05	0.64
Total	**Measured**	**0.14**	**8.5**	**4.7**	**0.1**	**0.07**	**0.02**	**0.09**
	Indicated	**2.15**	**195.4**	**139.4**	**3.0**	**1.51**	**0.36**	**1.31**
	Measured and Indicated	**2.29**	**204.0**	**144.2**	**3.1**	**1.58**	**0.39**	**1.40**
	Inferred	**0.44**	**37.9**	**104.9**	**0.4**	**0.27**	**0.07**	**0.78**

Note:

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

5. The cut-off grade calculation was based on the following assumptions: copper price of $3.00/lb, mining cost of $50/tonne, processing cost of $10/tonne, G&A and holding cost of $10/tonne, 90% copper recovery and 96% payable copper.

Table 14.13 Kipushi Zinc-Rich Bodies Measured and Indicated Mineral Resource Grade Tonnage Table, 14 June 2018

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

Note:

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

Table 14.14 Kipushi Zinc-Rich Bodies Inferred Mineral Resource Grade Tonnage Table, 14 June 2018

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

Note:

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

Table 14.15 Kipushi Copper-Rich Bodies Indicated Mineral Resource Grade Tonnage Table, 14 June 2018

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

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

Table 14.16 Kipushi Copper-Rich Bodies Inferred Mineral Resource Grade Tonnage Table, 14 June 2018

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

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

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 mineralisation has potential to continue at depth. The Mineral Resource occurs close to the DRC-Zambia Border and the Mineral Resource has been constrained to the area considered to be within the DRC.

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

14.12 Deleterious Elements

The grades of arsenic and cadmium were estimated as shown in Table 14.17

Table 14.17 Estimated Grades of Arsenic and Cadmium, 14 June 2018

Zone/ Class	Arsenic (%)	Cadmium (ppm)
Zinc-Rich Zones (Zn cut-off-grade 7%)		
Measured and Indicated	0.16	1,901
Inferred	0.25	1,540
Copper-Rich Zones excluding Fault Zone Splay (Cu cut-off-grade 1.5%)		
Measured and Indicated	0.30	202
Inferred	0.12	141
Fault Zone Splay (Cu cut-off-grade 1.5%)		
Inferred	2.94	1,548

14.13 Sulphide Percent Estimates

The sulphide grade of the Kipushi Mineral Resource was assigned to the block model for mining and metallurgical study purposes. The sulphide grades were calculated based on the copper, lead, zinc and sulphur grade estimates of the block model using the following methodology and assumptions:

- The proportion by weight of each metal in each mineral was calculated:
  - Chalcopyrite 34.643% Cu
  - Galena 86.622% Pb
  - Sphalerite 67.146% Zn
  - Pyrite 46.578% Fe
- The proportion by weight of sulphur in each mineral was calculated as follows:
  - Chalcopyrite 34.915% S
  - Galena 13.378% S
  - Sphalerite 32.854% S
  - Pyrite 53.422% S
- The ratio between sulphur and each metal was calculated:
  - S/Cu in chalcopyrite = 1.008
  - S/Pb in galena = 0.154
  - S/Zn in sphalerite = 0.489
  - S/Fe in pyrite = 1.147
- The total calculated sulphur grade for chalcopyrite, sphalerite and galena was assigned by dividing the metal grade by the respective sulphur metal ratio for copper, lead and zinc and added together.
- The total calculated sulphur grade for chalcopyrite, sphalerite and galena was subtracted from the ordinary kriged sulphur value to derive "excess sulphur", which was assigned to pyrite.
- The percentage of pyrite was calculated by dividing the "excess sulphur" grade by the proportion of sulphur in pyrite.
- The percentage of chalcopyrite was calculated by dividing the copper grade by the proportion of copper in chalcopyrite. The percentage of galena and sphalerite were calculated similarly.
- The calculated percent of each of the four sulphides was added together to provide an estimate of total sulphide in each block (CSULPHD in the block model). Any value greater than 100% was re-set to 100%.

There are a number of inaccuracies with this method:

- The sulphur/metal ratios assume theoretical values.
- All copper is assumed to be in chalcopyrite, although bornite and other copper minerals exist.
- Sphalerite is assumed to be in a pure form of ZnS. This is never the case and other elements such as iron will occur in the sphalerite.
- Pyrite occurs in the mineralised zones. The calculation assumes any sulphur not assigned to sphalerite, chalcopyrite or galena belongs to pyrite.
- Sulphur is regressed for some holes that did not have sulphur data, which tended to be Gécamines drillholes.
- The sulphur assigned to copper, lead and zinc can be more than the estimated sulphur grade. The negative "excess sulphur" grades were retained and used to calculate a pyrite value that was included in the total sulphide calculation.
- It is possible to calculate over 100% sulphides, when the zinc grade is very high. This occurred in 0.02% of the sub-blocks and in these cases the estimated sulphide grade was re-set to 100%.

Overall the QP considers that the total sulphide grade assigned to the block model is a reasonable approach in the absence of accurate data in which to estimate the sulphide grade from first principles.

15 MINERAL RESERVE ESTIMATES

This section has not been changed from the Kipushi 2017 Prefeasibility Study and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Kipushi 2017 PFS.

Access to the Kipushi Mine will be via the existing vertical shafts and internal decline to Big Zinc. Mining zones included in the current Kipushi mine plans occur at depths ranging from approximately 1,207 mRL down to 1,590 mRL with 0 mRL being the surface. The decline will be extended from the current position. Mined material will be trucked to the 1,150 mRL drive crusher tip, fed to the crusher on the 1,200 mRL, conveyed to silos for temporary storage, before being hoisted to the surface via Shaft 5.

The planned mining method for Kipushi is a combination of Sublevel Open Stoping (SLOS) and Pillar Retreat methods, at a steady-state mining rate of 0.8 Mtpa. The Big Zinc primary mining method is expected to be SLOS, with backfill. Mining will be performed using highly productive mechanised methods and CRF backfill will be utilised to fill open stopes. Depending on required composition and available material, excess waste rock, and DMS tailings will be used in the CRF mix as required.

Longhole stopes are 30 m high, separated by 15 m high sill pillars every 60 m, mined in a bottom up mining sequence. Stope back and wall support will not be required, provided an unfilled stope length of 60 m is not exceeded.

Remnant pillars containing singular 8 m x 6 m access drives will be staggered transversely to maximise ore extraction. Extracted sill pillars are not backfilled and will be left open for the Life-of-Mine (LOM). Scheduling ensures the pillars are not extracted until the stopes above and below the pillars are mined, backfilled and cured.

CRF of strength 1.2 MPa will be used for primary stope backfilling and 400 kPa for secondary stopes with no future exposure. Stockpiled surface waste material and DMS tailings will be transported via a 900 mm diameter borehole to an underground CRF mixing plant on the 1,320 mRL level. A surface cement plant will deliver cement slurry via a lined 380 mm diameter borehole to the underground CRF plant. A dedicated fleet of backfill trucks will transport the CRF from the underground CRF mixing plant to the stopes.

The optimised processing plant utilises dense media separation (DMS), followed by milling and a flotation recovery plant. 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 waste rock (predominantly dolomite) and mineralisation (sphalerite). The addition of milling and a flotation recovery plant resulted in an overall recovery of 89.6%, producing a consistent high-grade concentrate of 58.9% contained zinc. The improved concentrate grade results in lower transportation costs as compared to the Kipushi 2016 PEA. Net Smelter Return (NSR) is used to define the Mineral Reserve cut-offs, therefore cut-off is denominated in US$/t. By definition the cut-off is the point at which the costs are equal to the NSR. An elevated cut-off grade of $135/t NSR was used to define the mining shapes. The Marginal cut-off grade has been calculated to be $51/t NSR.

The Mineral Reserve estimate for Kipushi was based on the Mineral Resource which was first reported in the Kipushi 2016 PEA and was re stated in the Kipushi 2017 PFS. Only Measured Resources have been used for determination of the Proved Mineral Reserve and only Indicated Resources have been used for determination of the Probable Mineral Reserve. A zinc price of US$1.01 lb and a treatment charge of US$200 t concentrate were used in Mineral Reserve estimate. The zinc concentrate recovery and mass pull equations shown below. The economic analysis base case was prepared using a Zn cost of US$1.10 lb and a treatment charge of US$170 t concentrate.

- Zinc Concentrate Recovery (%) = $0.00000009 * Zn(Grade)^3 - 0.000004 * Zn(Grade)^2 + 0.0027 * Zn(Grade) + 0.831$.
- Mass Pull (%) = $0.017 * Zn(Grade) – 0.0583$.

A waste model was added to the resource model with matching prototype and parent cell size to ensure the accurate calculation of tonnage and dilution during optimisation. Optimal mineable stope shapes were created with only the measured and indicated tonnes considered. For preliminary reporting purposes block classification was analysed on a stope by stope basis, where dilution was assigned a classification dependent on the majority tonnage in the stope.

The Kipushi 2017 PFS Mineral Reserve has been estimated by Qualified Person Bernard Peters, Technical Director – Mining, OreWin Pty. Ltd., using the 2014 CIM Definition Standards. The Mineral Reserve is based on the January 2016 Mineral Resource. The effective date of the Mineral Reserve statement is 12 December 2017. Table 15.1 shows the total Proved and Probable Mineral Reserve of Kipushi.

Table 15.1 Kipushi Proved and Probable Reserve – Tonnage and Grades

Category	Tonnage (Mt)	Zn (%)	Contained Zn (kt)
Proved	3.10	35.41	1,098
Probable	5.48	30.29	1,660
Total	**8.58**	**32.14**	**2,758**

1. Effective date of the Mineral Reserves is 12 December 2017.
2. Net Smelter Return (NSR) is used to define the Mineral Reserve cut-offs, therefore cut-off is denominated in US$/t. By definition the cut-off is the point at which the costs are equal to the NSR. An elevated cut-off grade of US$135/t NSR (14.03% Zn) was used to define the mining shapes. The marginal cut-off grade has been calculated to be US$51/t NSR (3.43% Zn).
3. Mineral Reserves are based on a zinc price of $1.01/lb Zn and a treatment charge of $200/t concentrate.
4. Economic analysis to demonstrate the Kipushi 2017 PFS Mineral Reserve has used a zinc price of $1.10/lb Zn and a treatment charge of $170/t concentrate.
5. Only Measured Mineral Resources were used to report Proven Mineral Reserves and only Indicated Mineral Resources were used to report Probable Mineral Reserves.
6. Mineral Reserves reported above were not additive to the Mineral Resources and are quoted on a 100% project basis.
7. Totals may not match due to rounding.

15.1 Conclusion

Based on the mining production schedule and the criteria applied to the Kipushi Mineral Resource, the Proved and Probable Mineral Reserve has been demonstrated to be viable.

16 MINING METHODS

This section has not been changed from the Kipushi 2017 Prefeasibility Study and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Kipushi 2017 PFS.

16.1 Geotechnical

A geotechnical investigation was completed for Kipushi 2017 PFS based on 88 geotechnical borehole logs, 230 m of geotechnical scanline mapping and 60 structural borehole logs. Laboratory rock strength testing was also conducted to gain an understanding of the material properties across the project area. Geotechnical parameters based on strategies to manage the potential geotechnical risks have been derived and include backfill strength and support requirements.

16.1.1 Summary of Principal Objectives

The primary aims of the Kipushi prefeasibility underground geotechnical investigation and design were as follows:

- To increase the confidence in the mining geotechnical investigation conducted for the scoping study to the level of a PFS, by using additional geotechnical and structural data.
- To undertake numerical analyses based on the latest mine design and data from the mine site, to optimise the mine design going forward.
- To provide geotechnical mine design parameters for the Kipushi project to the level of a PFS.

16.1.2 Geotechnical Database

Geotechnical data in the form of geotechnical core logs from 88 boreholes were utilised to undertake rock mass classification for the Big Zinc orebody and surrounding rock. A list of boreholes utilised for this purpose is presented in Table 16.1.

To undertake the geotechnical structural analysis for the determination of the major discontinuity sets across the project area, 60 structural borehole logs were collated for use. (Table 16.2). In addition to this, SRK and Kipushi geologists carried out underground geotechnical scanline mapping on level 1220, the data from which was also used in the analysis (Table 16.3).

Table 16.1 Kipushi 2017 PFS Boreholes

Scoping			PFS				
KPU010	KPU057	KPU069	KPU006	KPU026	KPU039	KPU081	KPU093
KPU022	KPU058	KPU070	KPU011	KPU028	KPU041	KPU082	KPU093W1
KPU024	KPU059	KPU071	KPU012	KPU029	KPU043	KPU083	KPU094
KPU025	KPU060	KPU072	KPU013	KPU030	KPU044	KPU084	KPU095
KPU040	KPU061	KPU075	KPU014	KPU031	KPU045	KPU085	KPU096
KPU042	KPU062	KPU076	KPU015	KPU032	KPU047	KPU086	KPU097
KPU046	KPU063	KPU077	KPU016	KPU033	KPU049	KPU087	
KPU048	KPU064	KPU078	KPU017	KPU034	KPU052	KPU088	
KPU050	KPU065	KPU079	KPU018	KPU035	KPU053	KPU089	
KPU051	KPU066	KPU080	KPU019	KPU036	KPU054	KPU090	
KPU055	KPU067		KPU021	KPU037	KPU073	KPU091	
KPU056	KPU068		KPU023	KPU038	KPU074	KPU092	

Table 16.2 Kipushi Structural Boreholes

KPU003	KPU014	KPU036	KPU054	KPU067	KPU083
KPU004	KPU016	KPU037	KPU056	KPU068	KPU085
KPU005	KPU018	KPU038	KPU057	KPU069	KPU086
KPU006	KPU022	KPU040	KPU058	KPU072	KPU088
KPU007	KPU023	KPU042	KPU059	KPU075	KPU089
KPU008	KPU026	KPU044	KPU061	KPU077	KPU090
KPU009	KPU028	KPU046	KPU062	KPU079	KPU091
KPU010	KPU030	KPU051	KPU064	KPU080	KPU093
KPU011	KPU032	KPU052	KPU065	KPU081	KPU094
KPU013	KPU033	KPU053	KPU066	KPU082	KPU097

Table 16.3 Summary of Mapping Conducted

Rock Unit	Formation	Location	Total Length Mapped (m)
Dolomite	Upper Kakontwe	1N, 2N, 1SN, 2SN, D3	139.25
Dolomite	Middle Kakontwe	4SN, D2	64.50
Sphalerite	Big Zinc Ore Body	2SN, 4SN	13.80
Siltstone	Grand Lambeau	2SN, level 1270	12.70

During the Kipushi scoping study uniaxial compressive strength tests were carried out to gain an impression of the intact rock strength of the major lithological units in the project area. Following from this, a full suite of testing was implemented for the prefeasibility study, with the aim to gain further insight on the strength properties and variability of the Big Zinc orebody and its immediate hangingwall and footwall.

The laboratory testing programme comprised the following geomechanical tests:

- Uniaxial Compressive Strength with Young's Modulus and Poisson's Ratio (UCM).
- Uniaxial Compressive Strength (UCS).
- Uniaxial Indirect Tensile (BTS) Strength (Brazilian method).
- Base friction angle tests (BFA).

Samples selected for testing is summarised in Table 16.4.

Table 16.4 Samples Chosen for Laboratory Testing

Formation	Lithology	Source	No. of Samples				
			UCS	UCM	TCS	UTB	BFA
Upper Kakontwe	Dolomite (SDO)	KPU007, KPU065 KPU071, KPU091(W1), KPU093(W1)	0	5	9	5	3
Middle Kakontwe	Dolomite	KPU067, KPU071, KPU082, KPU050, KPU051, KPU055	6	10	15	9	8
Lower Kakontwe	Dolomite	KPU092, KPU093	0	5	9	5	4
Grand Lambeau	Siltstone (SSL), Sandstone (SST)	KPU065, KPU082, KPU083	0	4	6	5	5
Kipushi Fault Zone	Siltstone	KPU083	0	2	6	2	0
Big Zinc	Sphalerite	Level 1220 (BLOCK)	0	5	9	5	4

16.1.3 Geotechnical Model

Rock Properties

Rock mass properties were determined for the major lithological units across the project area derived from geomechanical tests that were conducted in RockLab, Pretoria. Lithologies that were tested were grouped based on stratigraphy and included:

- The upper, middle and lower dolomite (SDO) units (Kakontwe dolomite).
- Siltstone, sandstone, and shale from the Grand Lambeau/Kipushi fault zone (GLB/KFZ).
- Sphalerite from the Big Zinc (BZ) orebody.

A summary of the laboratory tests results for the major stratigraphic units are presented in Table 16.5.

Table 16.5 Laboratory Testing Results

Material Property	Rock Unit	GLB_KFZ	BZ	Kakontwe SDO
Density (kg/m^3)	Number of Tests	32	19	98
	Minimum	2.71	3.62	2.70
	Mean	2.78	3.88	2.85
	Maximum	2.86	4.03	3.03
	Standard Deviation	0.04	0.13	0.04
	Number of Tests	32	19	98
UCS (MPa)	Number of Tests	7	6	31
	Minimum	192	123	149
	Mean	261	238	278
	Maximum	315	326	343
	Standard Deviation	45	76	42
UTB (MPa)	Number of Tests	14	5	38
	Minimum	13	5	7
	Mean	17	7	12
	Maximum	20	10	15
	Standard Deviation	2	2	2
BFA (°)	Number of Tests	5	4	16
	Minimum	30	19	23
	Mean	35	26	34
	Maximum	38	29	42
	Standard Deviation	3	5	6

Material Property	Rock Unit	GLB_KFZ	BZ	Kakontwe SDO
Young's Modulus (GPa)	Number of Tests	2	1	18
	Minimum	70	75	70
	Mean	76	75	92
	Maximum	82	75	110
	Standard Deviation	8	-	10
Poisson's Ratio	Number of Tests	2	1	18
	Minimum	0.29	0.30	0.13
	Mean	0.29	0.30	0.37
	Maximum	0.29	0.30	0.58
	Standard Deviation	0.0	-	0.1
Hoek Brown	Number of Tests	22	12	88
	σ_{ci} – 1 std deviation	183	134	228
	Mean σ_{ci}	223	208	275
	σ_{ci} + 1 std deviation	263	281	322
	Standard Deviation	40	73	47
	mi	11	22	21

Following the analysis of the laboratory test results, rock strength (σ_{ci}) was classified based on mining position (hangingwall, orebody, and footwall). A summary of the strength of the hangingwall, orebody and footwall is presented in Table 16.6.

Table 16.6 Rock Strength (σ_{ci}) per Mining Unit (MPa)

Mining Position	No. of Geotechnical Intervals	Min	Mean	Max	Std Dev
5 m HW	54	160	272	326	34
HW	2658	52	255	326	30
OB	1054	124	253	326	41
FW	3152	134	273	326	12

Rock Mass Classification

To classify the quality of the rock mass use was made of Barton et al.'s (1974) Norwegian Geotechnical Institute's Q-System, Laubscher's 1990 rock mass rating (RMR) system and Hoek's (2013) Geological Strength Index (GSI) system, which was applied to each geotechnical interval for the 88 geotechnical boreholes located across the project area. The use of the Q-System was adopted to facilitate the derivation of Q' values for the stope design and for the determination of development support recommendations. Laubscher's RMR values were determined for the verification and validation of the Barton Q values derived for the various lithological units comprising the rock mass. The Geological Strength Index (GSI) was determined for the purposes of obtaining rock mass parameters for non-linear modelling, which may be required in the next stage of the project.

Rock mass classification results are presented from Table 16.7 to Table 16.10.

Based on the results of the rock mass classification, overall the rock mass at Kipushi may be classified as very good.

Table 16.7 RMR L90 Results

Mining Position	No. of Geotech. Intervals	Min	Mean	Max	Std Dev	*20 percentile*	*80 percentile*
5 m HW	48	46	81	100	15	68	99
HW	2658	29	81	100	18	63	100
OB	1054	35	87	100	17	69	100
FW	3152	32	72	100	17	58	96

Table 16.8 Q Results

Mining Position	No. of Geotech. Intervals	Min	Mean	Max	Std Dev	*20 percentile*	*80 percentile*
5 m HW	48	3	110	736	-	29	701
HW	2658	0.04	100	713	-	22	710
OB	1054	0.34	213	713	-	49	713
FW	3152	0.06	44	713	-	14	492
5 m FW	37	0.36	114	713	-	35	706

Table 16.9 Q' Results

Mining Position	No. of Geotech. Intervals	Min	Mean	Max	Std Dev	*20 percentile*	*80 percentile*
5 m HW	48	9	133	736	-	30	723
HW	2658	0.28	120	713	-	26	711
OB	1054	2.58	235	713	-	55	713
FW	3152	0.48	52	713	-	17	517
5 m FW	37	2.7	128	713	-	35	706

Table 16.10 GSI Results

Mining Position	No. of Geotech. Intervals	Min	Mean	Max	Std Dev	*20 percentile*	*80 percentile*
5 m HW	48	54	83	94	10	77	93
HW	2658	15	81	94	13	70	94
OB	1054	41	86	94	10	77	94
FW	3152	21	75	94	15	66	90

Geotechnical Block Model

Based on the results of the rock mass classification, a geotechnical block model was created for Kipushi Mine with the use of RMR values derived from Q based on Barton's equation (RMR = 15logQ + 50). This conversion was applied since Q values are expressed on a log scale and are thus difficult to statistically analyse. The aim of creating the block model was to provide a 3-dimensional impression of the rock mass conditions across the planned mining area.

Figure 16.1 illustrates the confidence in the block model, which decreases as the distance from the boreholes increase. As there is no data available in the far east of the project area note that this was not modelled. Sections through the Kipushi block model are presented in Figure 16.2.

Figure 16.1 Block Model Confidence (Plan View at 1,352 m)

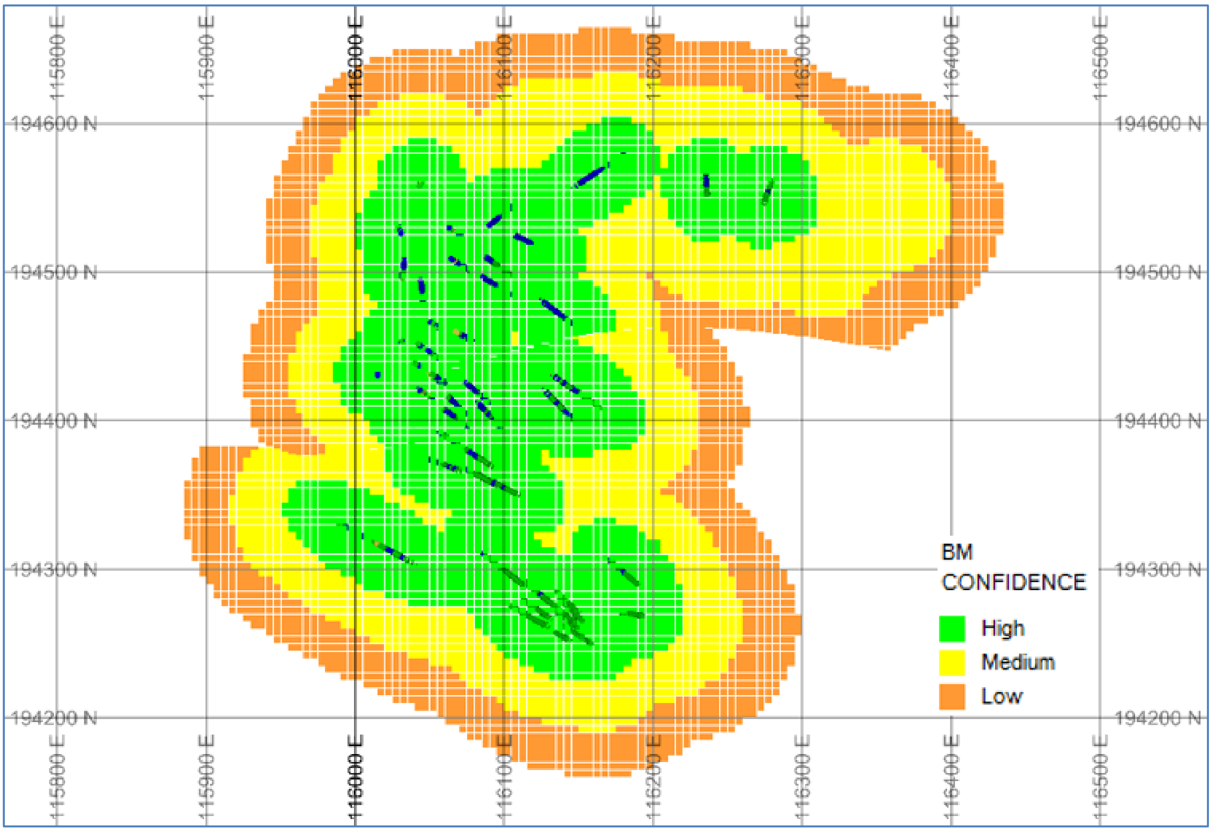
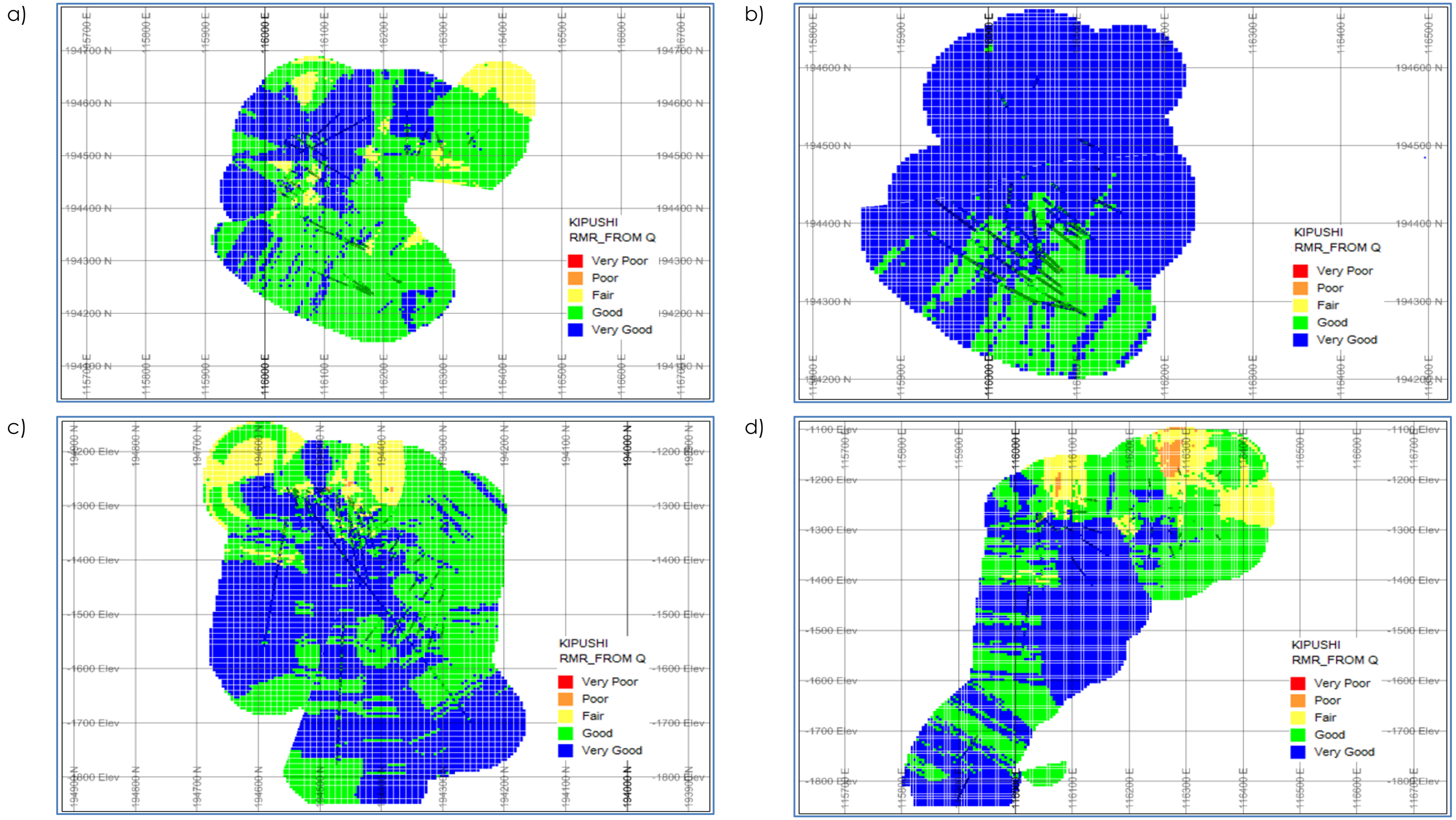


Figure 16.2 RMR from Q a) Plan View at 1307 m b) Plan View at 1507 m c) N-S Section Looking 116050 E d) W-E Section Looking 194553 N



From the creation of the block model the following was observed:

- The rock quality is lower in the north of the project area compared to the south.
- While this model provides insight on areas where potential instabilities can occur, it should not be used in a prescriptive manner to design rock support on a local scale.

Overall the geotechnical block model serves as a platform which can be built upon on a continuous basis as more data is gathered and as mining takes place. As there is no data present in various regions in the footwall of the project area, it is recommended that boreholes are drilled in these locations to verify the quality of the rock mass.

Structural Analysis

Based on the geology of the Kipushi region (), three geotechnical structural domains have been outlined, which comprises the major lithological units of the project area (Table 16.11). Using this classification, the Rocscience software DIPS was utilised to plot joint orientation data for each domain (Figure 16.3). Note that joint orientation data was plotted for the mapping and borehole data separately, and was then combined once trends were identified. A summary of the joint sets identified is presented in Table 16.12.

Table 16.11 Kipushi Structural Domains

Structural Domain	Major Lithologies
Kakontwe Dolomite	Upper Kakontwe dolomite (UK SDO) Middle Kakontwe dolomite (MK SDO)
Grand Lambeau/Kipushi Fault Zone (GLB/KFZ)	Siltstone (SSL) Shale (SSH) Sandstone (SST)
Orebody Material (Big Zinc)	Massive Brown Sphalerite (MBS) Massive Sulphides (MSM)

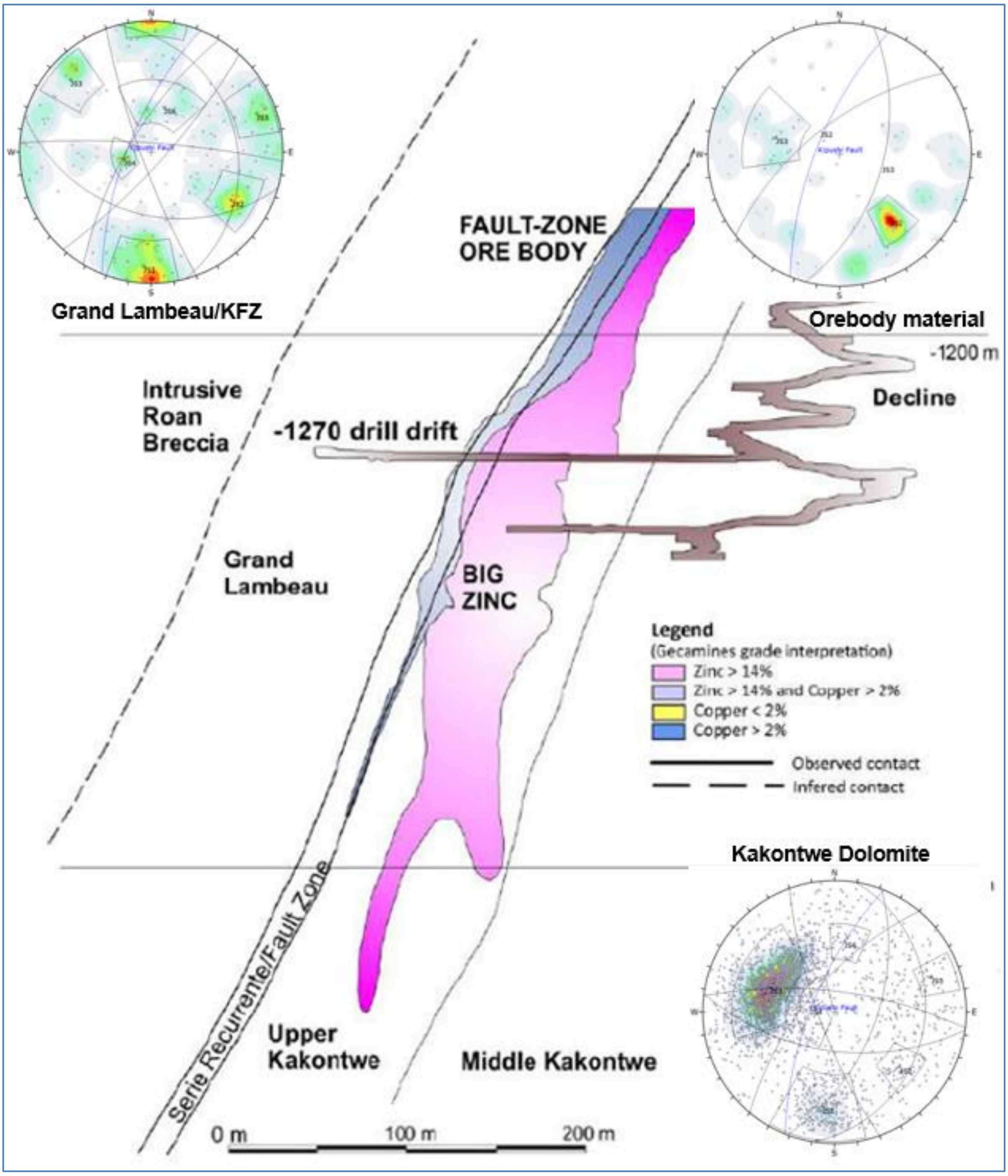
Table 16.12 Joint Set Summary

Joint set	Domain	Kakontwe SDO	GLB/KFZ	Orebody Material	Comment
JS1	Mean Dip	72	82	–	Major joint sets
	Mean Dip Direction	007	004	–	
JS2	Mean Dip	66	71	64	
	Mean Dip Direction	310	301	322	
JS3	Mean Dip	55	79	53	
	Mean Dip Direction	111	132	105	
JS4	Mean Dip	21	24	–	Non-dominant joint sets
	Mean Dip Direction	099	075	–	
JS5	Mean Dip	76	81	–	
	Mean Dip Direction	250	250	–	
JS6	Mean Dip	57	39	–	
	Mean Dip Direction	188	196	–	

Based on the structural analysis conducted for the Kipushi project, 6 joint sets have been identified across the project area. These sets are summarised as follows:

- JS1, JS2, and JS3 are the major joint sets identified in the project area.
- JS1 is a N to NNE steeply dipping set that represents the bedding of the sedimentary host rock.
- JS2 is a NW dipping set present across the project area. This joint set is sub-parallel to the Kipushi Fault.
- JS3 is a SE dipping set which is extremely dominant in the Kakontwe SDO.
- JS1 and JS2 are the major joint sets present in the GLB/KFZ.
- JS2 and JS3 are the major joint sets present in the Big Zinc orebody.
- JS1, JS2, and JS3 are the major joint sets present in the Kakontwe SDO.
- JS4 is a minor relatively flat dipping set that appears across the project area. Joint Set 5 and Joint Set 6 are non-dominant joint sets, which are present in localised zones.
- Overall joints are widely spaced, however there are zones of closely spaced joints that do occur which could locally influence stability. Joints are predominantly rough planar, and in most cases are only stained.

Figure 16.3 Identified Joint Sets per Structural Domain



16.1.4 Analysis and Design

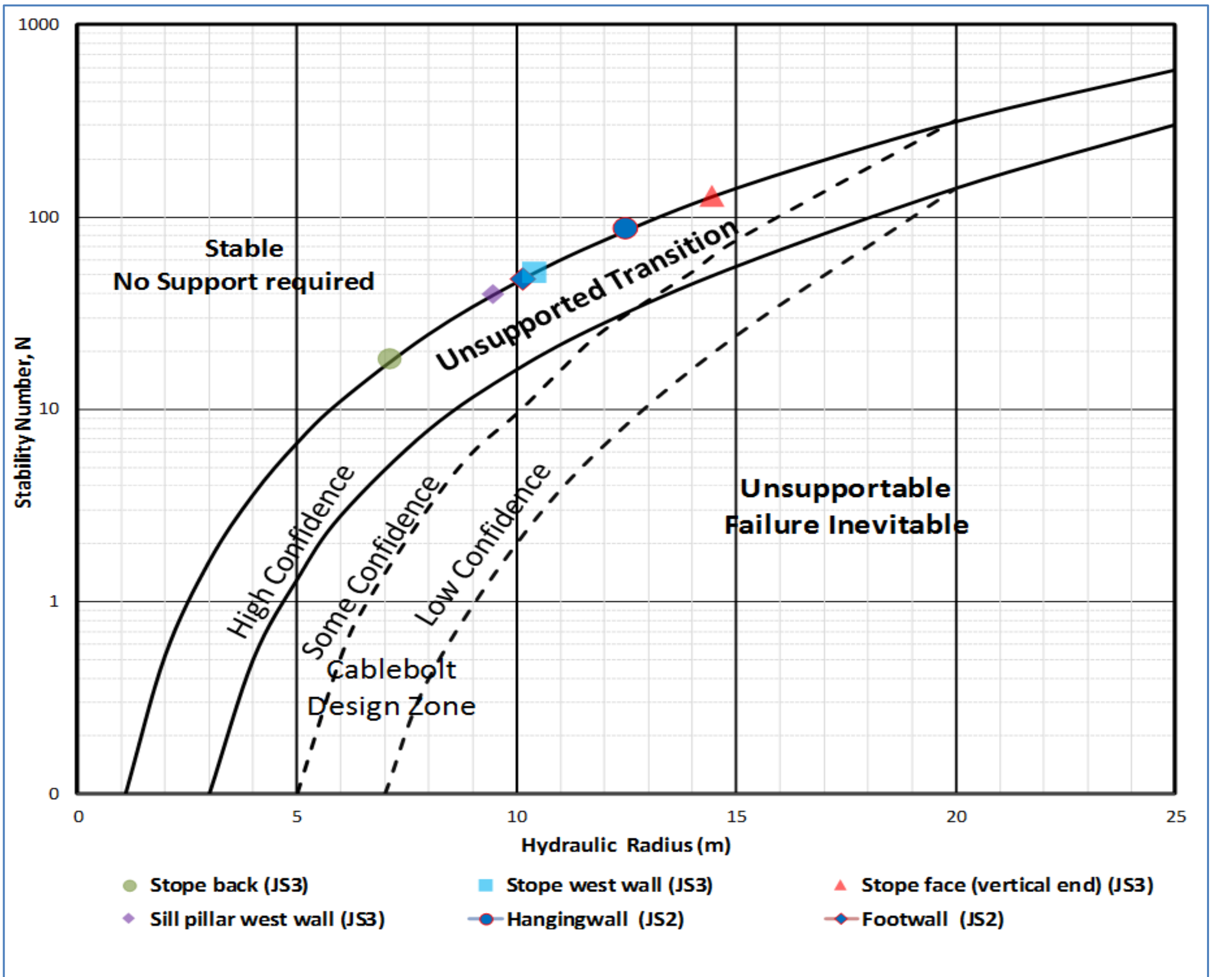
Empirical Stope Design

The stability of steeply dipping (greater than 60°) stopes was assessed using the stability method developed by Mathews et al. (1981) and modified by Potvin (1988), Potvin and Milne (1992) as presented by Hutchnison and Diederichs (1996).

To undertake the analysis, the stability number was determined from the rock mass classification carried out. The stability factors A, B, and C were obtained from the rock strength, stress analyses of the stopes and joint orientations in relation to the stope surfaces. Due to the depth of mining and mining induced stresses, the stress to strength ratios range between 0.25 and 0.61, despite the high strength of the rock mass. Figure 16.4 summarises the stability graph method input parameters and the allowable hydraulic radii for the stopes surfaces. Note that the analysis focused on the dominant joint sets whereby:

- JS2 and JS3 are the dominant joint sets for the orebody stope surfaces.
- JS1 and JS2 are dominant joint sets in the GLB/KFZ hangingwall.

Figure 16.4 Stability Assessment of Stope Sidewalls



	Q'	A	B	C	N	HR
Stope back (JS3)	55	0.3	0.7	2.0	18	7.1
Stope west wall (JS3)	55	0.4	0.4	5.7	52	10.4
Stope face (vertical end) (JS3)	55	0.4	1.0	5.7	129	14.4
Sill pillar west wall (JS3)	55	0.3	0.4	5.7	40	9.5
Hangingwall (JS2)	30	0.4	1.0	8.0	87	12.5
Footwall (JS2)	35	0.6	0.5	4.4	48	10.1

Table 16.13 summarises the maximum unsupported stope dimensions for the stope surfaces. Note that while the maximum HR per stope surface have been presented, the unsupported stope lengths are specific to practical considerations and the design HR, which is based on the stope dimensions as per the mine design parameters (Figure 16.5).

Table 16.13 Maximum Stope Dimensions (unsupported)

Stope Surface	Maximum unsupported Length (m)	Width (m)	Height (m)	Design HR (m)	Allowable HR (m)
Stope back (roof)	60	15	–	6.0	7.1
Stope walls (orebody)	60	–	30	10.0	10.4
Stope face (vertical end)		15	30	5.0	14.4
Stope walls (sill pillar)	60	–	15	6.0	9.5
Stope walls (hangingwall boundaries)	60	–	30	10.0	12.5
Stope walls (footwall boundaries)	60	–	30	10.0	10.1

The stope lengths for the hangingwall surface will be reviewed in the next stage of the project, where the SDO hangingwall and the GLB/KFZ hangingwall stope lengths (Figure 16.3) will be assessed and designed for separately. However, the design is not expected to change.

Assessment of the Shafts and Ventilation Raises

Planned ore passes and vent raises were assessed for raisebore stability for the PFS level of the project, where stability was assessed in terms of QR for each raise/orepass. This system accounts for the quality if the rock mass based on Barton's Q, and adjusts for the vertical walls, orientation of joints and weathering. Lithology was also considered in the assessment.

On the assessment of the location of each raise/orepass in relation to major lithologies, the following was observed:

- The collar elevations of ventilation raise 1 and ventilation raise 2 are located in the hangingwall (GLB/KFZ) whereby the raises traverse through the orebody to the dolomite in footwall.
- Ventilation raise 3 is located in the GLB/KFZ of the hangingwall.
- Ore passes 1 and 2 are located in the dolomite in the footwall.
- Where raises/ore passes traverse through different lithologies challenges with the drilling process may be experienced as a result of possible changes in rock strength or due to the contacts between lithologies.

To determine the quality of the rock in the location of the raises/orepasses, the geotechnical block model created for Kipushi was utilised. This block model is based on RMR values derived from Barton's (1974) Q system. Based on the information derived from the block model in the vicinity of the proposed raises and orepasses, QR was determined for each excavation (Table 16.14).

From the analysis it was observed that the QR values calculated indicate that the raises will be excavated in good to very rock.

Table 16.14 Raisebore Rock Quality (Q_R)

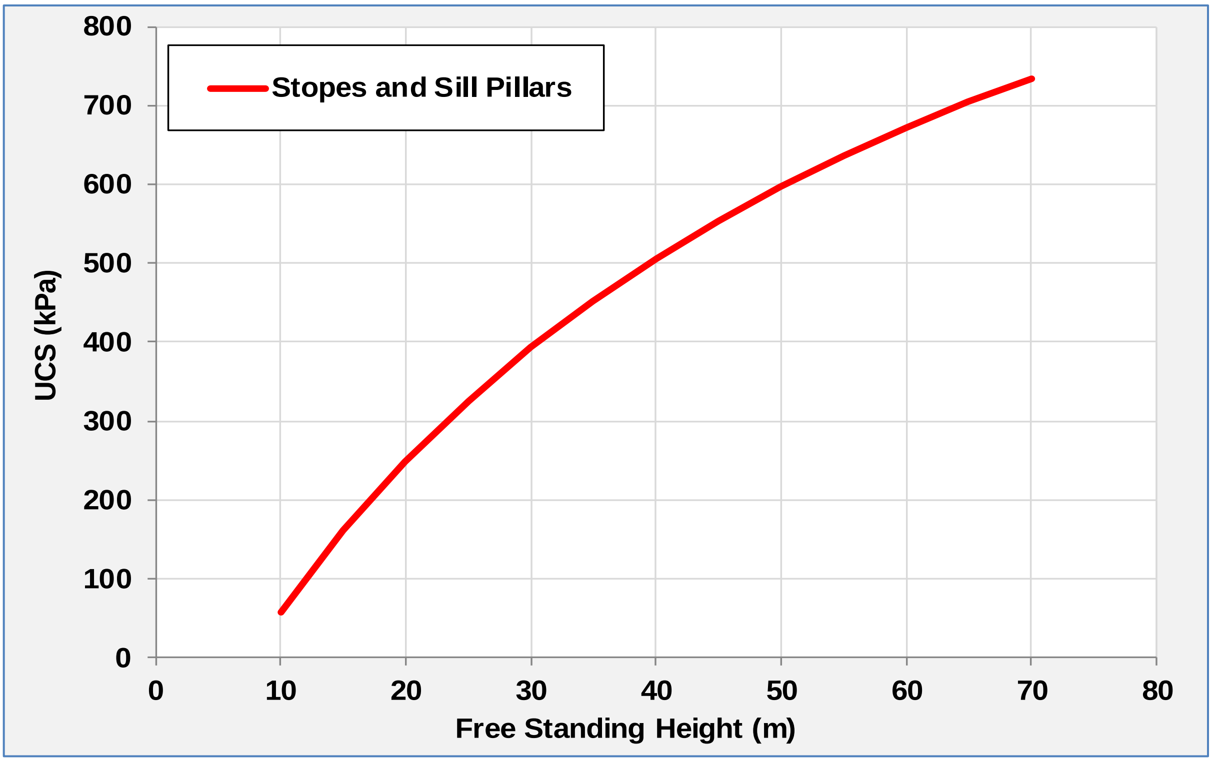
	Vent Raise 1	Vent Raise 2	Vent Raise 3	Orepass 1	Orepass 2
x co-ordinate	116147	116087	116013	116245	116217
y co-ordinate	194525	194538	194527	194422	194412
z_1 co-ordinate	-1155	-1290	-1440	-1320	-1440
z_2 co-ordinate	-1290	-1440	-1560	-1440	-1560
Diameter	4	4	4	4	4
Stratigraphy	GLB/KFZ/BZ/ SDO	GLB/KFZ/BZ/ SDO	GLB/KFZ	SDO	SDO
No. of blocks used in estimation	30	28	25	19	21
Block model confidence	Low to High	High	Medium to High	Low	Low
Q_R	**20**	**362**	**137**	**58**	**668**
Q_R rock quality	**Good**	**Very good**	**Very good**	**Very good**	**Very good**
MSUS (Maximum stable unsupported span in meters)	**9**	**18**	**18**	**15**	**18**

Backfill and Bulkhead Analysis

The extraction sequence of the primary and secondary stopes requires the use of backfill. The purpose of backfill is to maximise the extraction of the ore. It offers a working platform in the upper stopes and enables the extraction of secondary stopes and partial extraction of the sill pillar. It is important for the backfill to therefore be designed to be free standing for the extraction of primary stopes and for undercutting during sill pillar extraction.

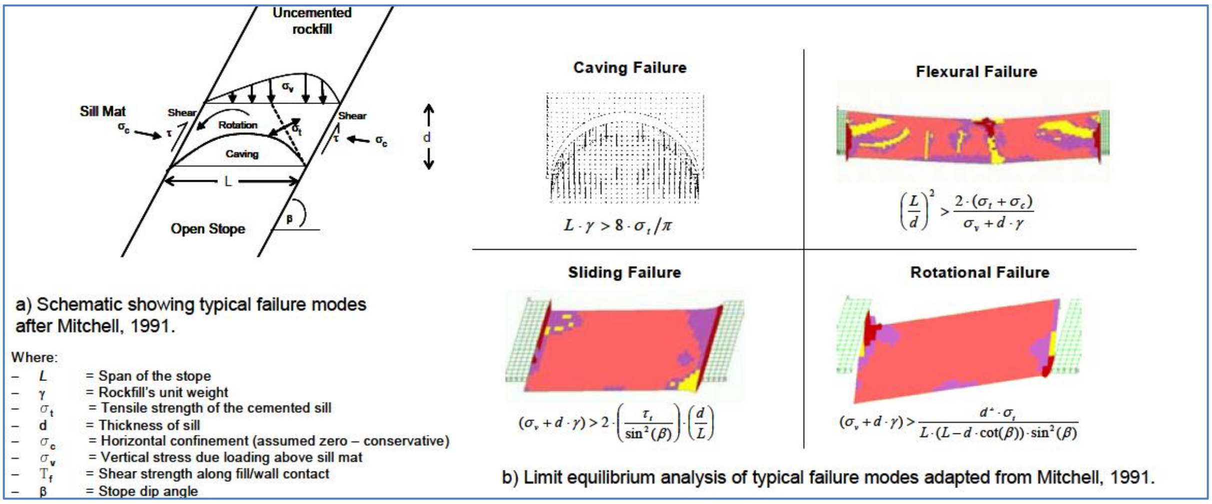
It is necessary to use sufficient binder to ensure that backfill is strong enough to be free standing when the walls are exposed during secondary stope extraction. Figure 16.5 shows the required backfill strength (400 kPa) to achieve a free-standing height in the upper primary stopes (30 m high and 15 m wide). The secondary stopes in a given sequence will not be exposed on one side and do not need to be designed to be free-standing. However, the backfill will require a minimum binder content of approximately 2% to prevent liquefaction.

Figure 16.5 Backfill Free-standing Height Strength Requirements



The extraction of the sill pillars following the mining of the lower primary stopes will require undercutting of the backfill in the stope above. Note that the backfill sill will be created on the lower primary stopes which are immediately above the primary sill pillars. The secondary sill pillar stopes are not planned for extraction. Backfill strengths therefore need to be designed to take the undercutting of the backfill sill into account. As backfill sills may fail under a number of failure mechanisms, it was necessary to consider possible failure mechanisms in the design. Figure 16.6 summarises potential backfill failure modes based on the limit equilibrium criteria established by Mitchell, 1999.

Figure 16.6 Limit Equilibrium Criteria (Mitchell, 1991)

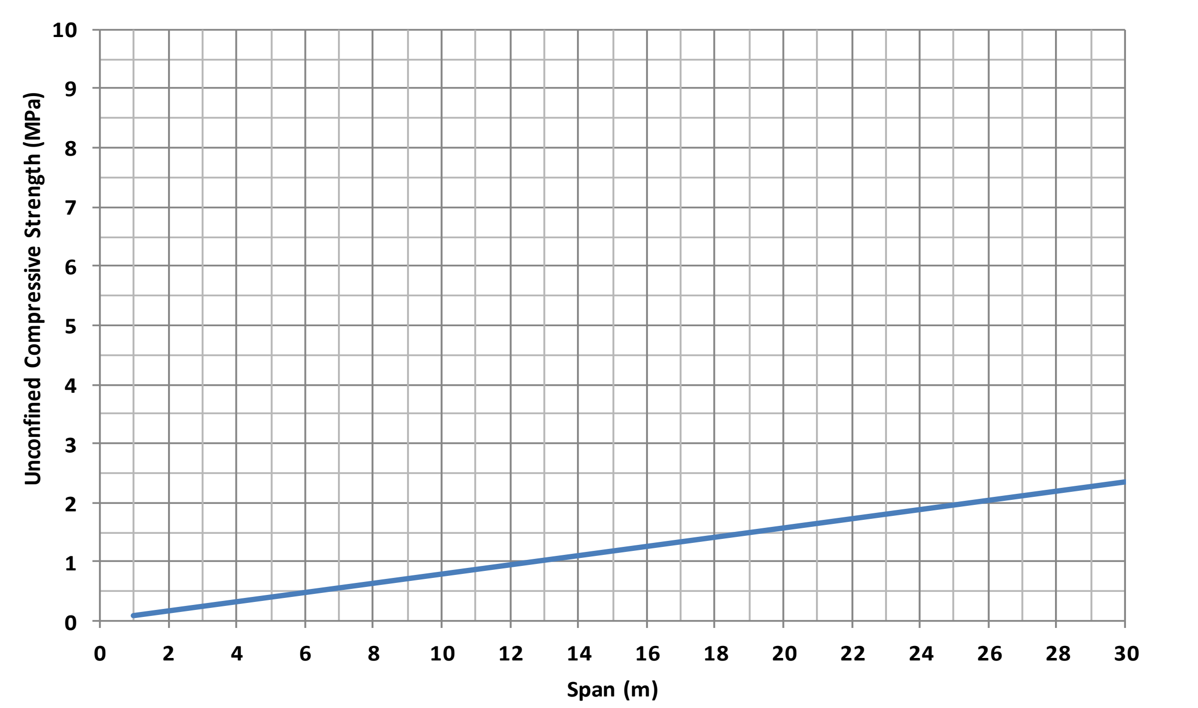


The backfill failure modes were investigated for vertical stopes using the limit equilibrium criteria. Stone (1993) suggested that for cemented rock fill, failure modes such as crushing, caving and sliding are generally negated when the sill thickness is greater than 0.5 times the span in the absence of the stope closure and when the unconfined compressive stress of backfill is greater than 1.5 MPa. However, backfill sill thicknesses of up to 30 m high will require compressive strength less than 1.5 MPa and hence it is important to test the failure modes.

Using the data and the same conditions described by Pakalnis et al. (2005), backfill failure modes were assessed to include a 30 m thick backfill sill.

Figure 16.7 show the strength requirements for the caving failure as described by Mitchell (1991) and Pakalnis et al. (2005) as the critical failure mode of a 30 m thick backfill sill will be caving. This will require a minimum sill strength of 1.2 MPa.

Figure 16.7 Stability Chart for Caving Failure



Bulkheads are constructed at the stope entrances and are designed to contain unconsolidated backfill material. The reinforced shotcrete arch bulkheads are recommended as they are relatively simple to construct. The bulkhead dimensions were obtained from planned tunnel dimensions (). Backfill properties were derived from literature (Golder associates, 2014 and Hugh et al., 2008) Bulkhead and backfill specifications are presented in Table 16.15.

Table 16.15 Bulkhead and Backfill Specifications

Bulkhead specification	**Value**
Maximum tunnel width	7 m
Maximum tunnel height	5 m
Plug length	6 m
Shotcrete compressive strength (at the time of backfill placement)	20 MPa
Backfill design parameters	Value
Saturated density	2.12 tonnes/m^3
Backfill friction angle	17°

The lateral load acting on the bulkhead was assumed to be applied by the backfill material. The load acting on the bulkhead is time dependent and reduces as the backfill material consolidates and cures. The rate of strength gain on the backfill at Kipushi Mine is currently unknown and it is recommended that instrumentation be considered to measure bulkhead loading during placement.

To determine the ultimate pressure (Wp) on the bulkhead analytical solutions by Johansen (1972) and Beer (1986) were utilised (Table 16.16). The analytical solutions assume a flexural mode of failure and perfect plasticity for the bulkhead.

Table 16.16 Analytical Solutions to Determine Ultimate Failure Pressure

Yield line solutions	Equation
All edges simply supported (Johansen, 1972)	$w_{p=} \frac{6\sigma_t h^2}{L^2}$
All edges fully supported (Johansen, 1972)	$w_{p=} \frac{12\sigma_t h^2}{L^2}$
Method by (Beer, 1986)	$w_{p=} \frac{3\sigma_c h^2}{L^2}$

Figure 16.8 shows the analytical solutions for a 7.0 m wide bulkhead. Figure 16.9 shows the factor of safety for the bulkhead for various backfilling lengths ranging from 0 m to 6 m for 0.2 m, 0.3 m, and 0.4 m shotcrete bulkhead. Under hydrostatic loading conditions a 6 m high tunnel with a bulkhead thickness of 0.4 will have a factor of safety of 1.6. Having 20 MPa shotcrete strength at the time of backfill placement a bulkhead thickness of 0.4 m is considered acceptable with a factor of safety of 1.6. Given the conservative nature of the bulkhead capacity estimation (Flat bulkhead) a bulkhead 0.4 m thick is recommended for the design.

Figure 16.8 Ultimate Barricade Load for Various Analytical Solutions

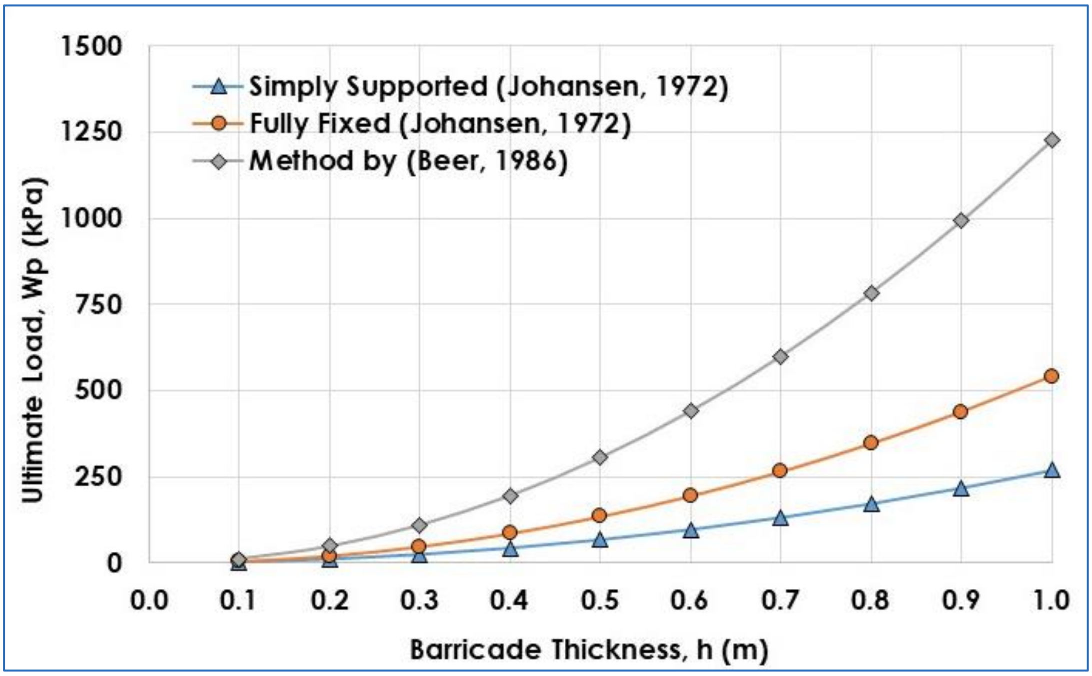
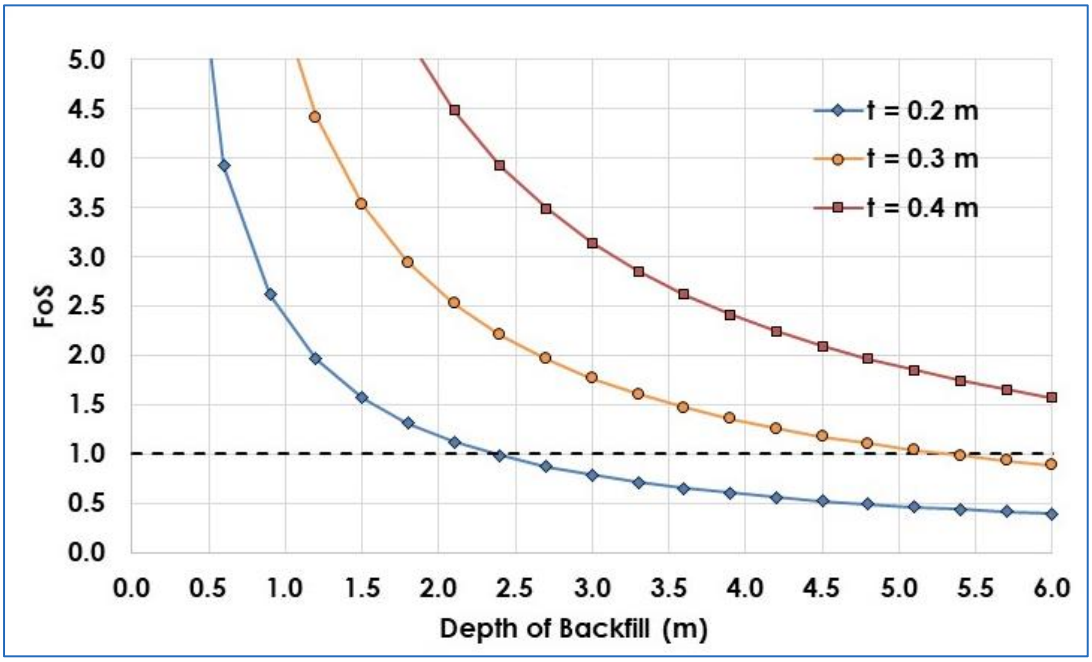


Figure 16.9 Factor of Safety for Fully Fixed Bulkhead Under Hydrostatic Loading Conditions



16.1.5 Geotechnical Design Parameters

Geotechnical design parameters have been derived based on the geotechnical properties determined, discussions held with the mine personnel and the preliminary elastic numerical modelling analysis conducted on the initial mine design. Geotechnical parameters have been outlined for the stope design and the mine access design and include backfill and support requirements.

In general, ground conditions in the project area are of a good quality and at this stage no major geological structures, which could adversely affect stability have been identified. The Kipushi fault is a major geological structure, however this fault is located in the hangingwall away from major development and geotechnical logs indicate that the quality of the rock in the vicinity of the fault generally ranges from fair to good. There are minor geological structures however, such as zones of closely spaced joints, which could locally influence stability.

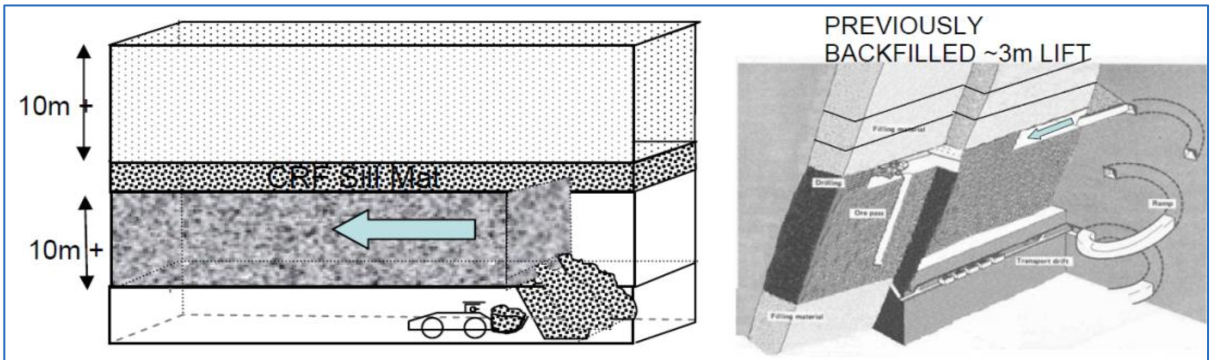
Stope Design Parameters

Longhole stopes have a total height of 60 m (comprising of an upper 30 m high stope and lower 30 m high stope) which will be separated by 15 m high sill pillars (Figure 16.14). The longholes stopes will be mined with a bottom up mining sequence whereby the lower stope is extracted first followed by the upper stope. The stopes will be extracted using a primary/secondary longhole stoping sequence with post filling (see Figure 16.30 and Figure 16.31). The sill pillars are expected to experience higher stress conditions and will require more geotechnical management. Mining is planned between level 1,207 m level and level 1,590 m.

Geotechnical recommendations for longhole stopes are as follows:

- Unfilled stope lengths should not exceed 60 m for the stope back and stope boundaries (hangingwall and footwall stopes).
- Stope back and wall support will not be required, providing the maximum dimensions are not exceeded.
- No entry is permitted in the stopes hence broken ore should be removed using remote Load Haul and Dump equipment (LHDs).
- Stopes should not be left open for long periods as rock conditions will deteriorate.
- Backfill strength requirements for primary stopes are provided in Section 16.1.4. Note that the first sub-level will always require greater backfill strength to allow for undercutting during the extraction of the sill pillar (Figure 16.10).
- It should be ensured that adequate time is planned for the construction of the bulkhead, backfilling of the stopes and curing of backfill to the required strength. This time should be incorporated into the stope scheduling.
- Additional binder will be required in the first sub-level stopes to achieve the required sill strength.
- A method of tight filling will be required to maximise the recovery of the secondary stopes (see Figure 16.25).
- Adjacent secondary stopes should not be mined simultaneously and backfill should be placed prior to mining the adjacent secondary stopes.

Figure 16.10 Example of Undercutting Backfill (Pakalnis, 2005)



a) Longhole mining under a cemented rock fill sill mat (Caceres, 2005)

b) Underhand cut and fill under a paste back.

The following geotechnical recommendations are specific to the extraction of sill pillars:

- The stope cross-sectional dimensions considered for sill pillar extraction at this stage are 15 m wide and 15 m high. The sill pillar will comprise primary and secondary stopes. Note that only primary stopes will be extracted. Secondary stopes will remain as permanent pillars.
- Due to the high stresses expected and the process of undercutting the backfill, stope/ore loss is likely during the extraction of the sill pillar.
- Lower production rates for the sill pillars should also be included in the scheduling due to challenges that may be faced and to take into account the time required to install additional support and the placement of backfill.

The following geotechnical recommendations apply to shafts and ventilation layouts:

- Stress damage on the existing and the proposed shafts will be minimal throughout the life-of-mine and standard shaft support will suffice (Table 16.19).
- The old shafts may require additional support to cater for time dependant deterioration of the rock walls.
- A few of the planned ventilation raises initially indicated that stress damage would be experienced (0.5 m to 1.5 m depth of fracturing) as these excavations were located approximately 10 m from the stopes. Ventilation raises have since been re-located and are now greater than 15 m from the stopes, and will experience minimal stress damage.

The following general rules apply for development layouts:

- Large, important excavations should be developed in good quality rock (Q>10) (87% based on current data) at least 20 m from stope excavations to avoid stress damage. Support requirements for large excavations is presented in Table 16.19.
- The decline will require S0 support from the level 1200 to 1410 (low stress environment) and S1 support from level 1410 to level 1590 (Table 16.18 and Table 16.19). S1 support will also be required above 1410 level, the ground conditions are poor (Q<4) (8% based on current data).
- Level, stope and parallel access drives should not be located less than 15 m away from the nearest stope to avoid stress interaction.
- Where major adverse geological structures occur, the excavation should be developed at a large angle (>45°) to the strike of the structure. Note that no major adverse geological structures have been identified at this stage.
- Stope drives will generally experience higher stresses than the access drives and rockbursts may occur. They will therefore require S2 support.
- Primary sill pillar stope drives will require S3 support as these are located in a high stress environment.
- S2 support should be installed in secondary sill pillar stope drives (this is since there will be no re-entry into these stope drives).
- Brow support will be required in the stope drives and the sill pillar stope drives (Table 16.19).

- Where secondary stope drives are developed prior to the extraction of the primary stopes, S2 support and rehabilitation (S3 support) of the secondary stope drives will be required.
- Where secondary stope drives are developed after the extraction of the primary stopes, S2 support will be required.
- Note that all tunnels must have an arched profile.

The extraction sequence of the stopes requires the use of backfill to safely mine the secondary stopes and sill pillars. Backfill and bulkhead requirements are outlined below.

The backfill strength requirement to achieve free standing heights in the stope and sill pillar stopes is 400 kPa (Figure 16.5). A detailed backfill sill analysis was conducted which has allowed for the optimisation of the design.

It is recommended that arched fibre reinforced shotcrete bulkheads are used to contain unconsolidated backfill. The bulkhead design is outlined in Section 16.1.4. The proposed bulkhead construction is illustrated in Figure 16.11. This will be implemented on the stope drives.

Figure 16.11 Construction of Fibrecrete Bulkhead (after Andrews et. al. 2010)

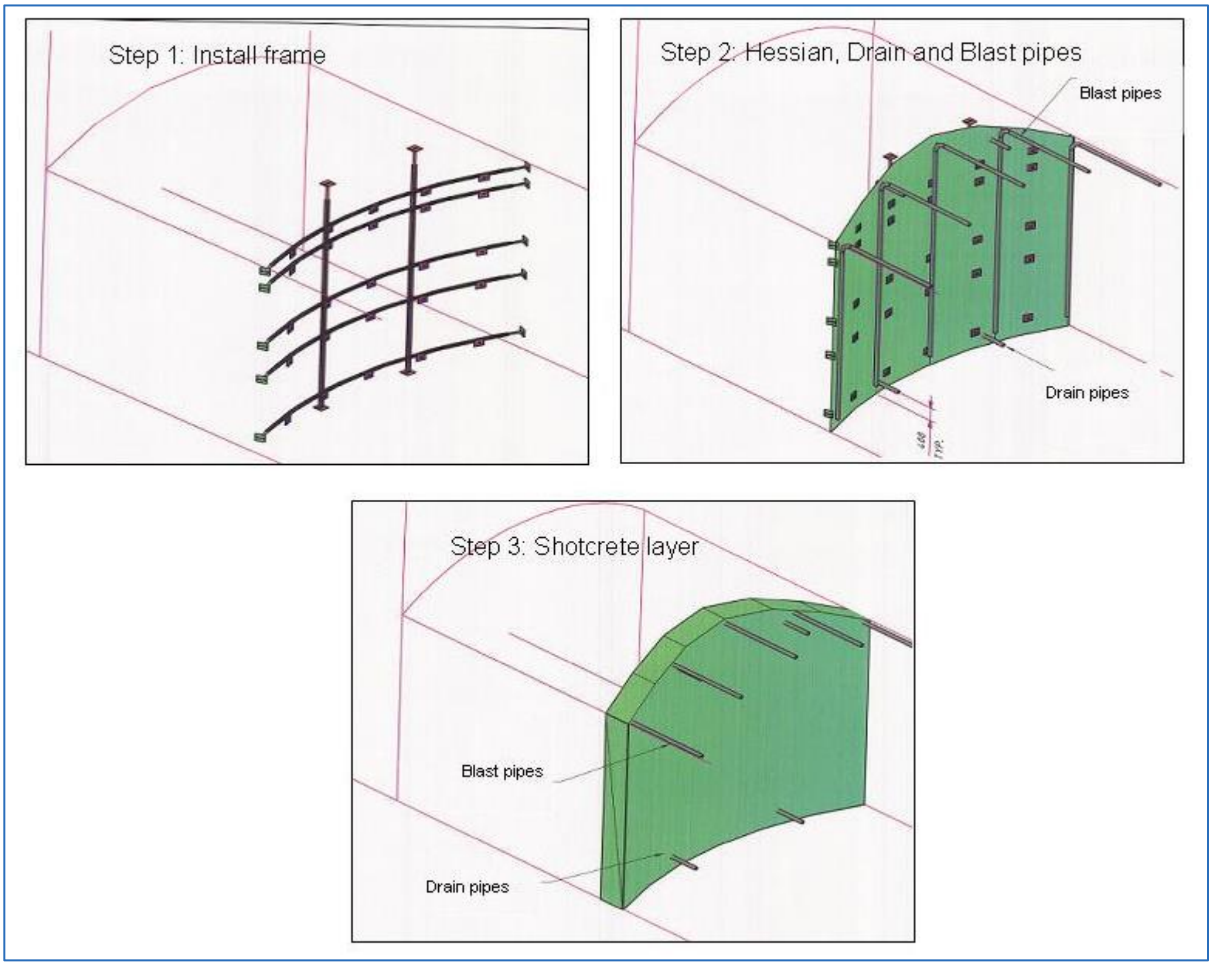


Figure 16.12 Example of Arched Wall Frame Kit

Support Requirements

As the Q values across the project area are generally high (), the depth of fracturing (DF) method (Martin et al, 1999, Cai and Kaiser, 2014) was used to determine the type of the support required (Table 16.17), which accounts for the influence of stresses.

Table 16.17 Support Requirements Based on DF

DF (m)	Damage Expected	Support
0 < DF	No fracturing	S0
0 < DF < 0.5	Minimum wall fracturing	S1
0.5< DF< 1.5	Stress damage, bulking and dynamic loading	S2
DF>1.5	Severe stress damage and dynamic loading	S3

The support requirements for the various excavations is presented in Table 16.18 and Table 16.19. Support specifications are presented in Table 16.20. Note that where the ground conditions are poor (Q<4) (8% based on current data), development support should be installed as specified in the development design parameters.

Table 16.18 Support Requirements per Excavation

Excavation	Level		Support Required
Decline (long term)	1207	1350	S0
	1350	1590	S1
Level access (medium term)	1207	1320	S0
Level access (medium term)	1320	1590	S1
Stope and parallel access (medium term)	1207	1400	S1
Stope and parallel access (medium term)	1460	1590	S2
Ore drives (short term)	1207	1590	S2
Sill Pillar Stope drives – Primary (short term)	1245	1590	S3
Sill Pillar Stope drives – Secondary (short term)	1245	1590	S2

Table 16.19 Support Standards for Excavations

Support	Area of Application	Support Standard
	Shafts (blind sink)	Primary support: Minimum 1.8 m long splits sets at 1.0 m x 1.5 m pattern with mesh. Secondary support: 300 mm concrete lining.
	Vent shafts (Raisebore) (high stress)	Minimum 50 mm shotcrete lining or concrete lining.
S0	Decline. No Fracturing. Access drive support for normal conditions (No fracturing). Geological structures.	Spot bolting. 2.4 m long tensioned rebar.
S1	Decline. Fracture depth <0.5 m. Access Drive. Fracture depth <0.5 m.	Primary support: 2.4 m long tensioned resin rebar in a 1.8 m x 1.8 m pattern with mesh in the crown down to the midway of the sidewall.
S2	Access Drive (Fracture depth between 0.5 m and 1.5 m). Stope drive support for high stress and dynamic conditions (Fracture depth between 0.5 m and 1.5 m). Stope drive (sill) support for secondary stopes (there will be no re-entry into these stope drives).	3 m long tensioned resin grouted yielding bars in a 1.2 m x 1.2 m pattern with mesh in the crown down to the 0.5 m from the floor.
S3	Stope drive (sill) support for primary stopes as extreme high stress and dynamic conditions are likely (Fracture depth >1.5 m).	3.0 m long tensioned resin yielding bar in a 1.0 m x 1.0 m pattern with 50 mm shotcrete and mesh across the drive, in crown and down to 0.5 m from the floor. Note that shotcrete should be applied first, followed by the installation of bolts and mesh.
	Stope brow support (where necessary).	Primary support + three rows (1.0 m apart) of three 6.5 m long grouted, cable anchors installed within 1.0 m of planned brow position.
	Support of intersections.	Primary support + minimum five 4.5 m long pre-tensioned, grouted cable anchors installed in the crown at the time of development.
	Support of large excavations (hoist chambers).	Primary support + pre-tensioned, grouted cable bolts (minimum length= half excavation span), maximum spacing = 0.5 x bolt length.

Table 16.20 Support Specifications

Support Type	Support Specification
Splits sets (SS 33)	Outer diameter 33.5 mm to 34.2 mm, yield strength 420 MPa black SUPRAFORM steel, minimum steel thickness 2.3 mm, hole size 30 mm to 32 mm.
Rebar	Yield strength 500 MPa black steel, 25 - 28 mm hole diameter, 20 mm bolt diameter hole size to match rebar diameter for resin mixing (maximum 4 mm annulus or effective mixing demonstrated through approved testing).
Yielding bar	Yield strength 500 MPa black steel, 25 - 28 mm hole diameter, 20 mm bolt diameter, minimum energy absorption 30 kJ within 300mm tunnel deformation, hole size to match resin mixing (maximum 4 mm annulus or effective mixing demonstrated through approved testing).
Shotcrete	Minimum 25 MPa shotcrete (28-day strength).
Cable anchor	Minimum 18 mm diameter black steel, 380 kN ultimate load.
Mesh	Black welded mesh, minimum 5 mm gauge, maximum 100 m aperture, blast resistant.
Capsule resin	Two component urethane silicate resin capsules. Fast <30sec and slow 5–10 min setting time.
Injection resin	Two component urethane silicate injection resin with water sealing properties.
Cable grout	Minimum 40 MPa Ordinary Portland Cement, water to cement ratio 0.35–0.40.

Monitoring

A 3D in-mine seismic monitoring system will be required to accurately gauge the rock mass response to mining. Approximately 24 x 14.5 Hz triaxial geophones should be installed approximately 100 m (or less) apart in 3D space to monitor the rock mass response of the sill pillars which are likely to become highly stressed as mining progresses.

In addition to seismic monitoring, the following will also be required:

- Closure monitoring with the use of closure meters/extensometers will be required to measure convergence (rock displacement) of the stope drives and vulnerable access drives. This will be extremely important during extraction of the sill pillar.
- Elastic modelling of mined and planned stopes as well as development tunnels should be carried out on a regular basis to determine where high stress concentrations are located. This will allow for updates to the planned design or for increased support to be applied in the affected areas.
- Stope Assessments should be conducted on a regular basis by the appropriate personnel (strata control officers, rock engineers etc.) where the rock mass conditions and the stability of the stopes are observed.
- Stope Reconciliation should be conducted based on assessments of dilution and the application of laser cavity monitoring scans. This will allow for an improved understanding of stope behaviour.

16.1.6 Elastic Modelling of the Final PFS Mine Design

Elastic modelling was initially carried out for the planned mining to determine geotechnical design parameters for Kipushi. Final modelling was then conducted to review the influence of stress on the access, stope development and the sill pillar. The planned and existing development at Kipushi is presented in Figure 16.13.

The elastic model was constructed using the Boundary Element program Map3D, whereby the extraction sequence used was provided by Ivanhoe and was carried out in six monthly steps. The elastic properties used in the model were obtained from the laboratory test results and rock mass classification. The potential damage on the main access levels, access ramps and stope access levels were assessed by calculating Depth of Fracturing (DF) (Martin, et al. (1999) and Cai and Kaiser, 2014).

From the modelling, the following was observed in the access tunnels and decline:

- The influence of stress on the decline during extraction of the stopes is negligible.
- The stress analysis indicates that the decline will initially require S0 support where there is no fracturing (depth of failure = 0 m) from 1220 to 1350 level. S1 support will be required from 1350 up to the lower levels.
- Stope and parallel access drives are placed approximately 20 m away from the stope and depth of fracturing between 1220 and 1410 level will be up to 0.5 m and therefore S1 support will be adequate. Where these tunnels intersect geological structures additional S2 support will be required to manage ground conditions.
- The depth of fracturing for stopes access drives below 1410 level is up to 1.0 m and S2 support will required.
- Stress analysis indicates that the lower primary stope ore drives will require S2 support. The lower secondary stope ore drives will experience stress damage during the extraction of closing pillars and will require additional support.
- Tunnel intersections will require additional intersection support.
- The upper stope ore drives will experience stress damage during the extraction of the lower stopes (depth of fracturing greater than 1.5) and could potentially influence the mucking operations even though these are short term excavations. Rehabilitation of the ore drives may therefore be required in some circumstances.
- The primary and secondary sill pillar ore drives will experience stress damage with depth of failure greater than 1.5 m. S3 support will be required for the primary sill pillar ore drives. The secondary ore drives are only required for drilling purposes and backfilling the stope below. S2 support for the secondary sill pillar ore drives will be adequate for the short term. However, where conditions deteriorate rapidly additional support may be installed if access is still needed.
- Stope brow support may be required.

The following was observed for the ore passes and ventilation raises:

- The orepasses are located approximately 30 m from the stope abutments. The raises will experience a minor stress damage throughout the life-of-mine (depth of failure <0.1 m). This depth of failure is considered negligible and should not influence the stability of the orepasses.
- The ventilation raises are located in the hangingwall (relatively high stress environment) approximately 30 m from the stope edges and will experience some damage from the Year 2027 up to the end of the life-of-mine (depth of failure approximately 0.35 m). Stress damage in the ventilation raises is unlikely to affect the function of these excavations.

16.1.7 Findings and Recommendations

A summary of the findings and recommendations for the mining geotechnical investigation is presented below:

- Overall the rock is very strong and very little stress damage has been observed. With increasing depth and the influence of stope abutments, stress damage can be anticipated in future. The rock is strong and brittle and therefore rockbursts are likely to occur when the anticipated stress damage is significant.
- As there is no data present in various regions in the footwall of the project area, it is recommended that boreholes are drilled in these locations to verify the quality of the rock mass.
- Where possible, further geotechnical mapping should be conducted at lower levels in the mine to improve the statistical distribution of the dataset and improve confidence in the joint sets. Acoustic televiewer (ATV) logging data from boreholes has also been provided for this purpose and will be utilised in FS level of the project.
- To determine the quality of the rock in the location of the raises/orepasses, the geotechnical block model was utilised. Based on this evaluation it was observed that the raises/orepasses are located in good to very good rock. However, as the raises are sometimes located in areas where there is low confidence in the block model, it is recommended that a borehole is drilled and geotechnically and structurally logged for each raise.
- The extraction of the stope requires the use of backfill to provide a working platform in the upper stopes, support of the stope surfaces and protection cover during the extraction of the primary sill pillar stopes. Based on the evaluation the following will be required:
  - To achieve a free-standing height in the upper primary stopes (30 m high and 15 m wide) a backfill strength of 400 kPa will be required.
  - Backfill sill will be required in the lower primary stopes. The secondary stope sidewalls in a given sequence will not be exposed hence binder is not required in the cemented rock fill.
  - The critical failure mode for a 30 m backfill sill is caving failure. A backfill sill strength of 1.2 MPa for the lower stopes will be required for a 30 m high and 15 m wide stope to enable undercutting.

- The shotcrete strength of 20 MPa at the time of backfill placement and thickness of 0.4 m with a factor safety 1.6 for a maximum tunnel height of 6.0 m will be required for the bulkhead design.

16.2 Mining

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

MSA provided the May 2017 resource model using the January 2016 Resource Model (fkigmod_23-01-2016.dm), reporting that it had the same mineral resource to the January 2016 model. The May 2017 model includes a % Sulphides field to report the dolomite. The model was checked and comparisons were made to ensure continuity between the resource reports and model. Parent cell size in the model is 5 m E x 5 m N x 5 m RL and sub cells are also included.

The Mineral Reserve estimate for Kipushi was based on the Mineral Resource reported in the Kipushi 2016 PEA. Only Measured Resources have been used for determination of the Proved Mineral Reserve and only Indicated Resources have been used for determination of the Probable Mineral Reserve.

A waste model was added to the resource model, with matching prototype and parent cell size, to ensure the accurate calculation of tonnage and dilution during optimisation. Optimal mineable stope shapes were created using Mineable Shape Optimiser (MSO) and Datamine with only the measured and indicated tonnes considered.

Sill pillar placement and the locations of the sill pillars in the mineable shape are crucial to determine the optimum extracted tonnes with the highest grade. The sill pillar was positioned at different levels between the -1,230 mRL and the -1,290 mRL and it was found that located on the -1,260 mRL level was optimal.

Using BDT10 the marginal cut-off grade was calculated to be approximately $51/t NSR10. Ore in development above this cut-off was then reclassified as either Low-Grade (LG) or High-Grade (HG). LG ore was classified as greater than $51/t NSR10 and less than $135/t NSR10 and HG ore was classified as greater than $135/t NSR10.

The planned mining method is a combination of Sublevel Open Stoping (SLOS) and Pillar Retreat methods, at a steady-state mining rate of 0.8 Mtpa. The Big Zinc's primary mining method is expected to be SLOS, with CRF backfill. The sill pillars are expected to be mined using the Pillar Retreat mining method once the adjacent stopes are backfilled.

The existing mining infrastructure consists of five surface vertical shafts and a number of sub-vertical shafts allowing access to deeper levels.

The 850 mRL will be utilised as intermediate level on the Shaft 5 to allow personnel and equipment to enter the mine workings, without doing so via the main haulage and crusher level, minimising interactions and downtime to the haulage network.

The main working area is connected to Shaft 5 via the 1,150 mRL main haulage level. There is a crusher chamber at 1,200 mRL; the crusher level is now dewatered. The underground infrastructure exposed since dewatering, is in relatively good order. The crusher is being replaced as the cost of refurbishment was determined to exceed the replacement cost.

Figure 16.13 Planned and Existing Development at Kipushi

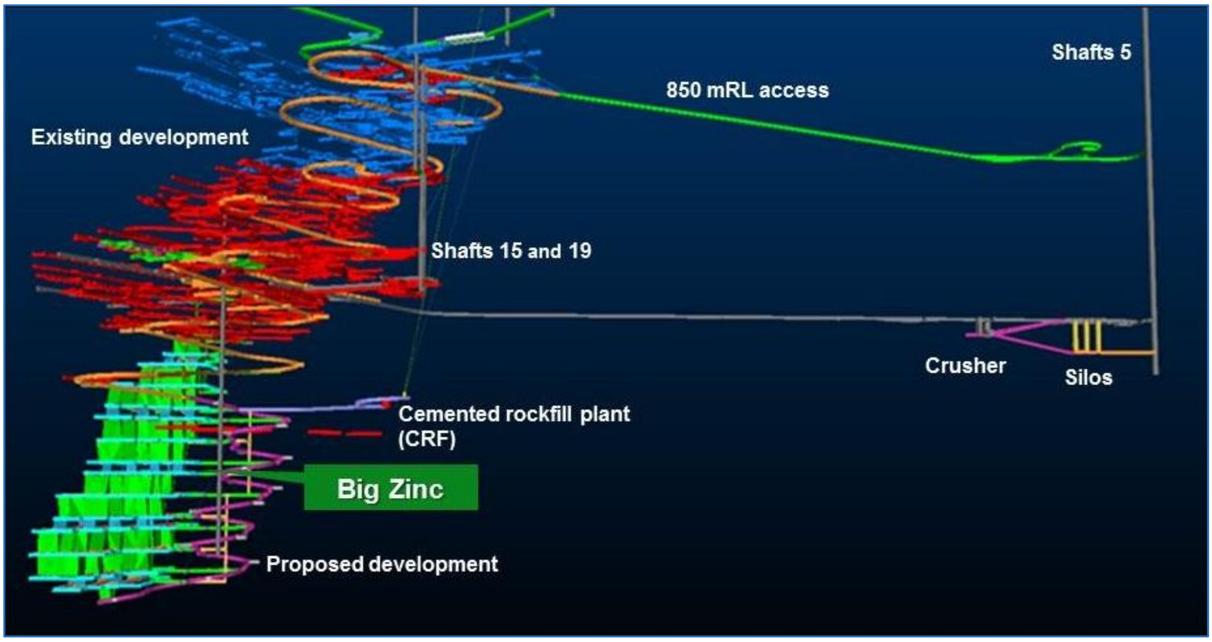


Figure by Ivanhoe, 2016.

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.

A network of underground pumps, cascading dams and pipework currently dewaters the mine at a maximum rate of 3,500 m^3/h.

Workshops and magazines exist on the 1,132 mRL and 850 mRL levels. These areas require rehabilitation but will provide locations for machine maintenance, breakdown areas, welding bays, wash bays, tyre changing and storage, explosives storage, lubricant tanks, and diesel storage.

Mine access will be via the existing shafts and internal decline to Big Zinc. The decline will be extended from the current position. Mined material will be trucked to the 1,150 mRL drive crusher tip, fed to the crusher on the 1,200 mRL and then conveyed to silos for temporary storage before being hoisted to the surface up Shaft 5.

Support classifications have been developed for all underground applications including decline, stope drives, sill pillar stope drives and access drives. The mine production schedule was used to calculate support requirements over the Life-of-Mine (LOM).

Once drilled and blasted, material from stopes and pillars will be transported by LHD to either ore or waste stockpiles on each level. From there, material will be trucked to the 1,150 mRL level crusher tip, for crushing and then hoisting to surface via Shaft 5. Excavated stopes will then be backfilled with CRF trucked from the CRF plant on the 1,320 mRL level.

The Big Zinc is located at depths ranging from approximately 1,185–1,710 mRL with the Kipushi 2017 PFS focused on the 1,185–1,590 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 stockpiles and then loaded into trucks. From active mining levels the trucks are expected to haul material to the 1,150 mRL crusher tip.

Longhole stopes are 30 m high which will be separated by 15 m high sill pillars every 60 m and mined with a bottom up mining sequence. Stope back and wall support will not be required provided an unfilled stope length of 60 m is not exceeded.

Figure 16.14 Big Zinc Stope Cross-section

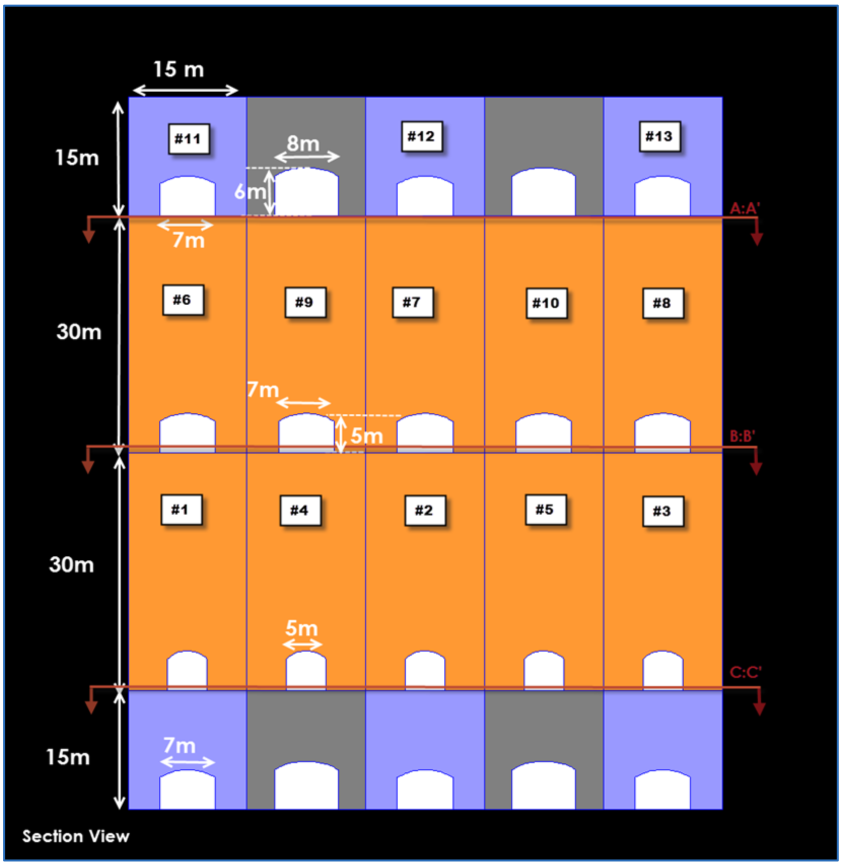


Figure by OreWin, 2017.

Remnant pillars containing singular 8 m x 6 m access drives will be staggered transversely to maximise ore extraction. Extracted sill pillars are not backfilled and will be left open for the LOM. Scheduling ensures the pillars are not extracted until the stopes above and below are mined, backfilled and cured.

CRF of strength 1.2 MPa will be used for primary stope backfilling and 400 kPa for secondary stopes with no future exposure. Stockpiled surface waste material and DMS tailings will be transported down a 900 mm borehole, to an underground CRF mixing plant on the 1,320 mRL level. A surface loader will feed an aggregate screen and conveyor, which will sort and supply waste material to the waste pass at the required rate.

Fuel is supplied via a 1,325 m long fuel line, from the surface to the 850 mRL workshop and from the 850 mRL workshop to the 1,332 mRL workshop. Fuel will be piped to hose reel stations for underground equipment refuelling. The fuelling station will have the storage tanks and pumps installed in an enclosed drift with fire doors and appropriate fire suppression systems.

The equipment requirements for the Kipushi project are split into two categories, fixed equipment and mobile equipment. The equipment requirements for each category cover the major components for the operation.

The mobile equipment required for lateral development includes drill jumbos, LHDs, haul trucks, and ground support equipment. Mobile equipment required for stoping includes longhole drill rigs, LHDs, haul trucks, and ground support equipment.

Due to the historic nature of Kipushi and the fact it is currently under care and maintenance, significant underground fixed equipment exists in place. This includes shaft winders, skips and cages, workshop facilities, silos, conveyors and dewatering pumping infrastructure.

The existing crusher chamber and accompanying excavations on the 1,150 mRL at Kipushi are currently being rehabilitated and will be recommissioned. The existing Crushing and Ore handling infrastructure will be replaced.

The site personnel are provided partially by the client and partially by the contractor. Both provide a combination of expatriates and nationals. The expatriates are employed at the beginning of the project, to be replaced by nationals as the project goes on. The client provides labour for roles from the surface down to and including the crusher, while the contractor provides labour from the crusher down to the face.

The estimated peak airflow requirement for Kipushi is 570 m^3/s. The airflow requirements are based on meeting the minimum regulatory airflow requirements for diesel exhaust dilution. With the shafts available as airways at Kipushi, the exhaust configuration options will be twin exhausts on Shafts No. 4 and No. 3. Peak primary fan operating pressure is over 4,000 Pa and centrifugal fans are recommended.

The Kipushi ventilation design uses a combination of parallel and series ventilation activities. Primary exhaust is provided on each level. More polluting activities, such as production mucking and backfill, should be parallel ventilated on the level direct to exhaust. The remaining less polluting development and non-diesel activities can either be parallel ventilated, or series ventilated from the decline.

Pumping requirements were based on a hydrogeological study which shows the simulated mine inflow rates predicted for the 2016 PEA designs. The mining rate and design depth for the 2016 PEA exceeded the current rate and designs and such, the predicted inflows were used to estimate the inflow for the purposes of this study. Based on the specifications, two centrifugal dewatering pumps would be required on the 1,290 mRL, pumping to the 1,112 mRL dam. When mining reached the 1,440 mRL the pumping station would be moved to this level.

16.2.1 Introduction

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

All pertinent technical and economic data related to the mining of the resource was provided by Ivanhoe. All dollar amounts throughout the report are expressed in 2017 US Dollars (US$).

16.2.2 Mining Block Model

MSA provided the May 2017 resource model using the January 2016 Resource Model (fkigmod_23-01-2016.dm), reporting that it had the same mineral resource to the January 2016 model. The May 2017 model includes a % Sulphides field to report the dolomite. The model was checked, and comparisons were made to ensure continuity between the resource reports and model. Parent cell size in the model is 5 m E x 5 m N x 5 m RL and sub cells are also included.

The Mineral Reserve estimate for Kipushi was based on the Mineral Resource reported in the Kipushi 2016 PEA. Only Measured Resources have been used for determination of the Proved Mineral Reserve and only Indicated Resources have been used for determination of the Probable Mineral Reserve.

16.2.2.1 Base Data Template 10

A Base Data Template 10 (BDT10) was created in MS Excel, calculating the Net Smelter Return 10 (NSR10) on a $/% basis before adding to the model. This allowed calculation of the marginal and break-even cut-off grades and meant ongoing checks could be performed during the optimisation process. In the model, NSR10 was calculated on a block by block basis using testwork algorithms, formulae, prices, recoveries and costs shown in Table 16.21 and Table 16.22. Each block was assigned a dollar per mined tonne value NSR10.

Table 16.21 NSR10 Modifying Factors

Maximum Concentrate Zinc Grade	%	60.58
Maximum Zinc Recovery	%	95.00
Maximum Mass Pull	%	95.00
Payable Zinc Metal	%	85.00
DMS Concentrate Recovery Constants		
Testwork Zinc tail grade	%	10.76
Testwork Zinc feed grade	%	45.36
Zn non-floating	%	0.10
Concentrate Moisture Content	%	12.00
Zinc Metal Price	US$/lb	1.01
Treatment Charge	US$/t dmt	200.00
Concentrate Transport Cost	US$/t conc.	249.61
DRC Royalty	% of smelter payables	2.00
Gécamines Royalty	% of smelter payables	2.50
DRC Export Tax	% of the value of the export	1.00

Table 16.22 Zinc Recovery and Concentrate Algorithm

Zinc Concentrate Recovery	%	$0.00000009 * Zn_{Grade}^3 - 0.000004 * Zn_{Grade}^2 + 0.0027 * Zn_{Grade} + 0.831$
Mass Pull	%	$0.017 * Zn_{Grade} - 0.0583$
Concentrate Zinc Grade	%	$\frac{Zn_{Recovery} * Zn_{Grade}}{\text{Mass Pull}}$
Tail Zinc Grade	%	$Zn_{nf} \frac{+ (Zn_{tail\ ref} - Zn_{nf})}{Zn_{ref\ Grade}} * Zn_{Grade}$

The following conditions were applied to the recovery, mass pull, concentrate grade and tailings grade formulae to ensure continuity for the range of zinc values in the model.

- Where the calculated mass pull is less than 0.0% the mass pull is 0.0%.
- Where the feed grade is less than 60.58% Zn and the calculated concentrate grade is greater than 60.58% Zn the concentrate grade is 60.58% Zn.
- Where the feed grade is greater than 60.58% Zn the concentrate grade equals the feed grade.
- Where the mass pull exceeds 95%, the tailings grade equals the feed grade.

Elemental smelter penalties were not included in the NSR10 calculation.

16.2.2.2 Optimisation

A waste model was added to the resource model with matching prototype and parent cell size to ensure the accurate calculation of tonnage and dilution during optimisation. Optimal mineable stope shapes were created using Mineable Shape Optimiser (MSO) and Datamine with only the measured and indicated tonnes considered. For preliminary reporting purposes block classification was analysed on a stope by stope basis, where dilution was assigned a classification dependent on the majority tonnage in the stope. This was to ensure continuity between the initial optimisation reports in Datamine and the schedule reporting in Enhanced Production Scheduler (EPS). The optimisations were run at $10 increments between $100/t NSR10 and $180/t NSR10 using the parameters shown in Table 16.23.

Table 16.23 MSO Optimisation Parameters

Stope height	m	30
Pillar height	m	15
Pillar spacing	m	60
Stope/Pillar width	m	15
Stope/pillar length	m	60

Table 16.24 shows the extraction and dilution values used for the calculation of mined tonnes in the stopes and pillars.

Table 16.24 Stope and Pillar Extraction and Dilution

Mined	Extraction (%)	Dilution (%)
Stope	90	2.5
Pillar	59	20

The intention of the optimisation was to define a reserve with a mined tonnage of approximately 8.6 Mt at 32.00% Zn, giving a mine life of 10 years at a rate of 800 ktpa.

The $135/t optimisation shape is shown in Figure 16.15, where the stopes can be seen in green and the pillars in blue. The creation of half height, underhanging and outlying stopes and pillars by MSO required manual checks of the stope shapes. Any shapes that were un-mineable were removed or extended to reduce complications during the design process.

Figure 16.15 Updated $135/t NSR10 MSO Optimisation

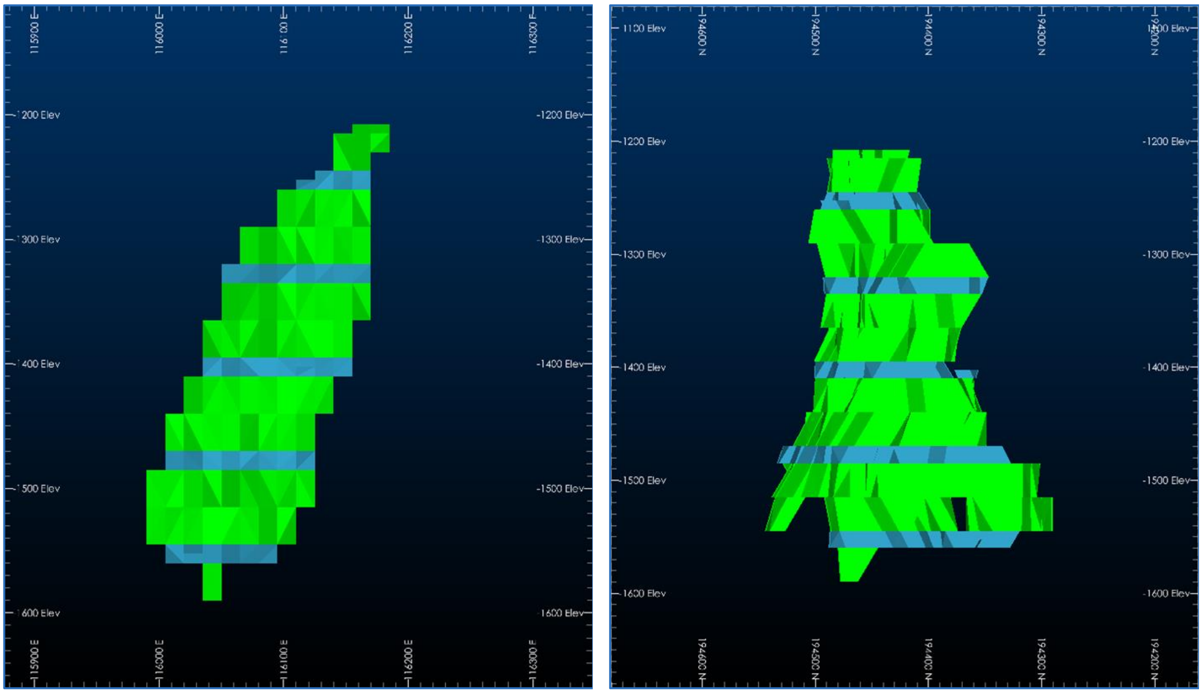


Figure by OreWin, 2017.

Re-calculated tonnage for the updated $135/t optimisation shape was compared with the initial optimisations to ensure it still met the reserve requirements and is shown in Table 16.25. The zinc grade tonnage curve can be seen in Figure 16.16.

Table 16.25 Preliminary MSO Optimisation with Updated $135/t

Inventory	In-situ Tonnes (kt)	Mined (kt)	NSRDMS+CU (BDT10) ($/t)	Zn (%)	Cu (%)	S (%)	Fe (%)	Sulphide (%)
2016PEA Design	9,413	8,293	306.84	31.90	0.49	23.25	7.98	63.00
$100/t NSR10 Cut-off	10,867	9,577	287.30	29.89	0.57	22.35	8.00	60.25
$110/t NSR10 Cut-off	10,554	9,297	293.19	30.50	0.57	22.71	8.08	61.27
$120/t NSR10 Cut-off	10,241	9,017	299.15	31.12	0.55	23.09	8.17	62.33
$130/t NSR10 Cut-off	9,921	8,738	304.98	31.72	0.54	23.42	8.22	63.29
$135/t NSR10 Cut-off	**9,713**	**8,554**	**308.12**	**32.04**	**0.53**	**23.61**	**8.25**	**63.81**
$140/t NSR10 Cut-off	9,651	8,496	310.30	32.27	0.53	23.72	8.27	64.16
$150/t NSR10 Cut-off	9,340	8,228	316.25	32.88	0.52	24.09	8.34	65.19
$160/t NSR10 Cut-off	9,015	7,939	322.79	33.56	0.51	24.45	8.41	66.25
$170/t NSR10 Cut-off	8,674	7,643	329.91	34.29	0.51	24.87	8.49	67.45
$180/t NSR10 Cut-off	8,381	7,383	335.95	34.91	0.50	25.20	8.55	68.42

Figure 16.16 Grade Tonnage Curve %Zn - Updated $135/t

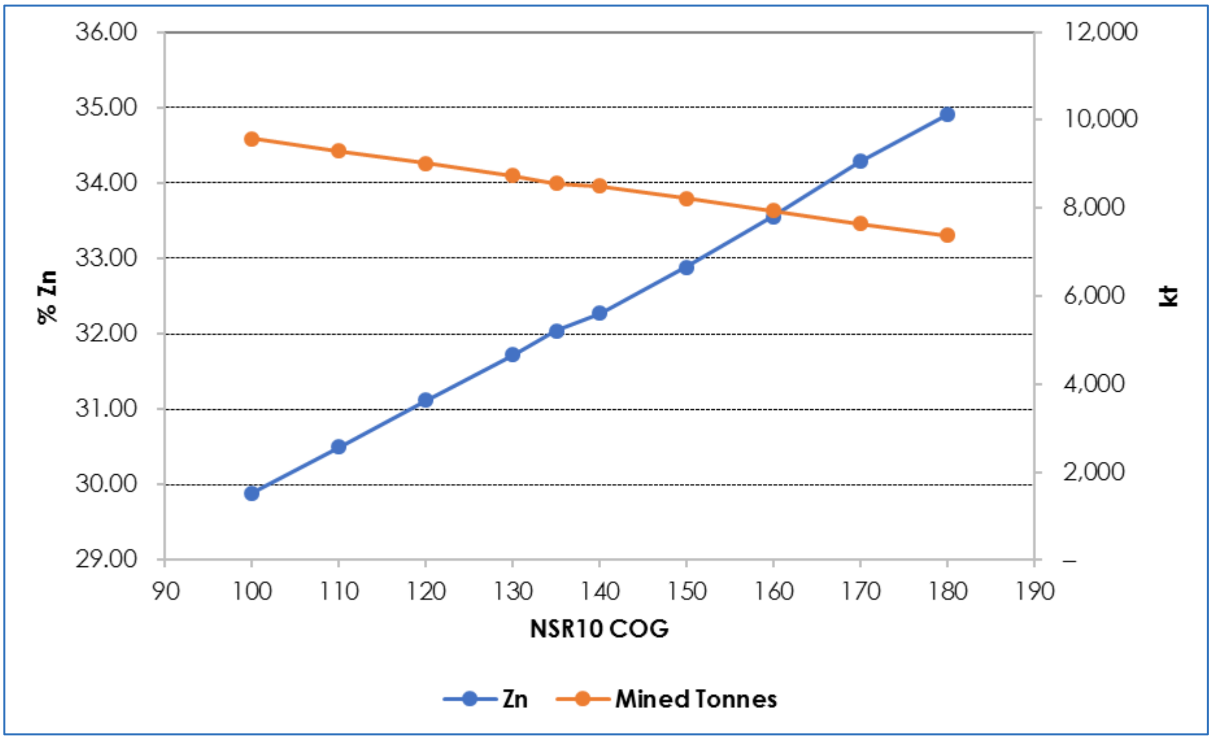


Figure by OreWin, 2017.

16.2.2.3 Pillar Placement

Sill pillar placement and the locations of the sill pillars in the mineable shape are crucial to determine the optimum extracted tonnes with the highest-grade. The sill pillar was positioned at different levels between the -1,230 mRL and the -1,290 mRL and it was found that the -1,260 mRL level was the optimal location. Figure 16.17 and Table 16.26 show the tonnes and grades by level with the optimal crown and sill pillar configuration.

Figure 16.17 -1,260 mRL Optimised Pillar Location

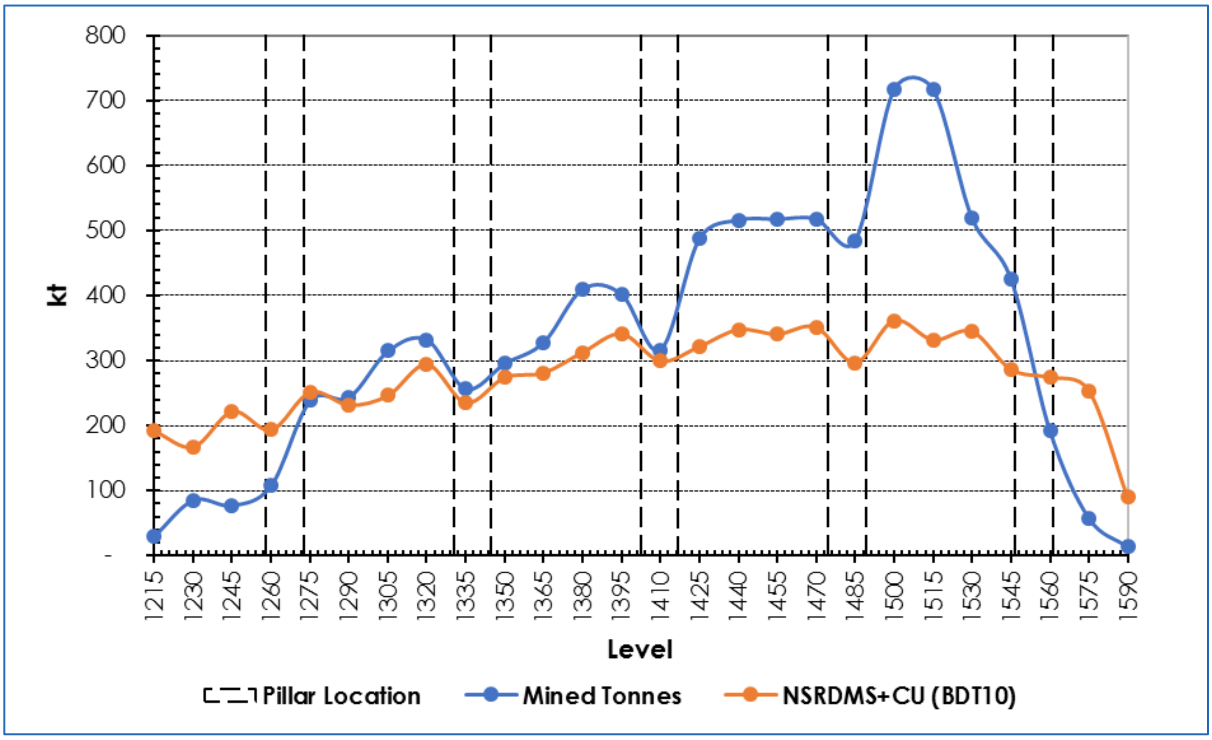


Figure by OreWin, 2017.

Table 16.26 1,260 mRL Optimised Pillar Location

Level	15 m Level Type	Extraction (%)	Dilution (%)	In-situ Tonnes (kt)	Mined Tonnes (kt)	NSRDMS+CU (BDT10) ($/t)	Zn (%)
1215	Stope	90.00%	2.50%	33	30	191.62	20.41
1230	Stope	90.00%	2.50%	93	85	167.36	17.65
1245	Stope	90.00%	2.50%	72	67	231.72	24.45
1260	Pillar	59.00%	20.00%	147	104	195.48	20.59
1275	Stope	90.00%	2.50%	258	238	251.49	26.35
1290	Stope	90.00%	2.50%	252	233	235.53	24.69
1305	Stope	90.00%	2.50%	330	305	249.74	26.28
1320	Stope	90.00%	2.50%	360	332	294.04	30.74
1335	Pillar	59.00%	20.00%	357	253	236.93	24.76
1350	Stope	90.00%	2.50%	362	334	252.21	26.33
1365	Stope	90.00%	2.50%	358	330	280.05	29.25
1380	Stope	90.00%	2.50%	445	410	312.60	32.54
1395	Stope	90.00%	2.50%	436	402	340.90	35.39
1410	Pillar	59.00%	20.00%	447	317	299.17	31.04
1425	Stope	90.00%	2.50%	529	488	321.48	33.40
1440	Stope	90.00%	2.50%	560	516	347.37	36.04
1455	Stope	90.00%	2.50%	561	518	340.54	35.35
1470	Stope	90.00%	2.50%	562	518	351.13	36.38
1485	Pillar	59.00%	20.00%	684	484	296.52	30.72
1500	Stope	90.00%	2.50%	777	717	361.50	37.44
1515	Stope	90.00%	2.50%	778	718	331.60	34.45
1530	Stope	90.00%	2.50%	551	508	348.40	36.15
1545	Stope	90.00%	2.50%	462	426	286.45	29.79
1560	Pillar	59.00%	20.00%	260	184	278.74	28.93
1575	Stope	90.00%	2.50%	26	24	337.33	35.00
1590	Stope	90.00%	2.50%	14	13	89.92	9.31
Total				**9,713**	**8,554**	**308.12**	**32.04**

16.2.2.4 Preliminary Development Tonnes

Depending on cut-off grade and shape parameters, the MSO optimisation algorithm selects most, but not all, tonnes above cut-off in the model to create a minable shape. This results in blocks above cut-off grade lying outside the stope shapes, which have mineable potential during the development phase.

Using BDT10 the marginal cut-off grade was calculated to be approximately $51/t NSR10. Ore in development above this cut-off was then reclassified as either Low-Grade (LG) or High-Grade (HG). LG ore was classified as greater than $51/t NSR10 and less than $135/t NSR10 and HG ore was classified as greater than $135/t NSR10.

Preliminary mine designs were used to approximate the potential tonnage of LG and HG ore in development. Table 16.27 and Figure 16.18 shows the tonnes, grade and location of the development ore outside the $135/t NSR10 MSO shapes. EPS scheduling classification of the development as either ore or waste ensured the LG and HG tonnes were either considered waste, stockpiled or sent to the plant.

Table 16.27 Preliminary Development Tonnes

Class	In-situ Tonnes (kt)	Mined (kt)	NSRDMS+CU (BDT10) ($/t)	Zn (%)	Cu (%)	S (%)	Fe (%)	Sulphide (%)
Measured	–	–	–	–	–	–	–	–
HG Development (NSR10>$135/t)	17	17	221.65	23.43	0.62	22.08	10.42	55.89
LG Development ($51/t>NSR10<$135/t)	25	25	89.26	8.33	0.75	11.77	7.36	27.88
Subtotal	**42**	**42**	**143.45**	**14.51**	**0.70**	**15.99**	**8.61**	**39.35**
Indicated	–	–	–	–	–	–	–	–
HG Development (NSR10>$135/t)	72	72	229.14	24.24	0.79	20.83	9.22	54.47
LG Development ($51/t>NSR10<$135/t)	85	85	86.62	7.99	1.52	11.73	7.89	28.40
Subtotal	**157**	**157**	**152.07**	**15.45**	**1.19**	**15.91**	**8.50**	**40.37**
Total	**200**	**200**	**150.24**	**15.25**	**1.08**	**15.93**	**8.52**	**40.15**

Figure 16.18 Preliminary Development Tonnes

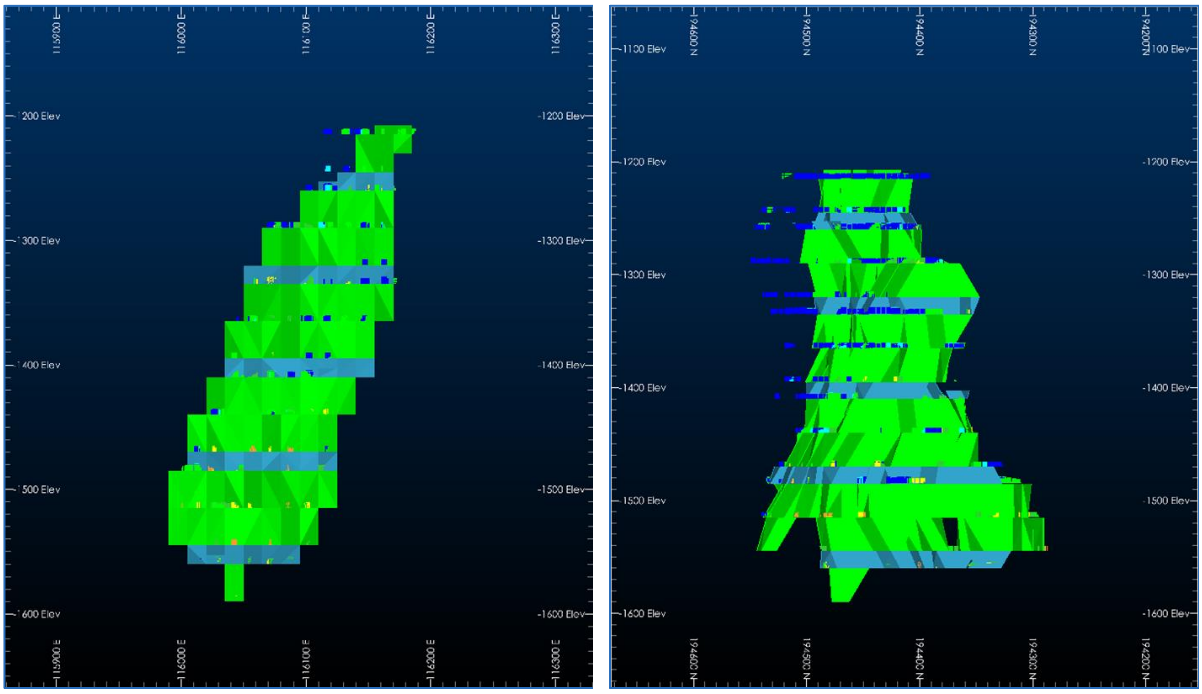


Figure by OreWin, 2017.

16.2.2.5 Final Stope and Pillar Design

Due to geotechnical design constraints the secondary pillars and remaining half height stopes were removed. Knowing the development size in the remaining permanent sill pillar and the primary sill pillar, extracted pillars were determined based on contained zinc metal. Figure 16.19 shows the final stope and pillar shapes that were used for scheduling.

Figure 16.19 Final Stope and Pillar Shapes

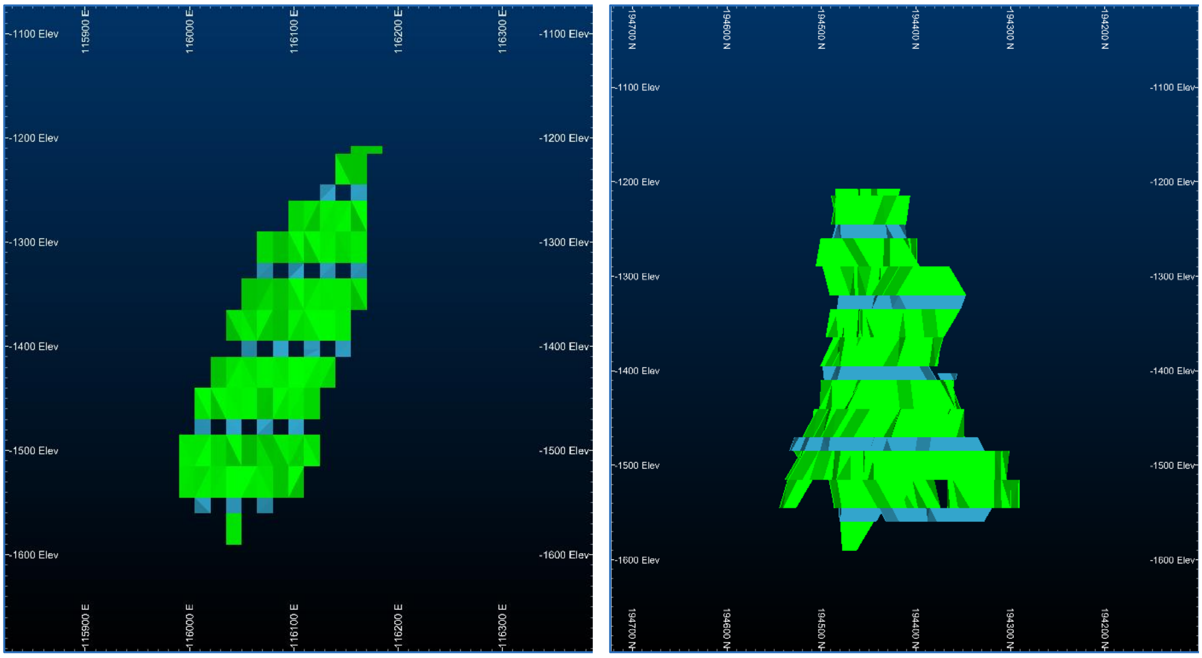


Figure by OreWin, 2017.

16.3 Mining Method Selection

The key criteria considered in the selection of the mining method for Kipushi include the following.

- Maintain maximum productivities.
- Minimise ramp-up period by developing mining zones as early as possible.
- Maintain high overall recovery rates.
- Minimize overall dilution.

The planned mining method is a combination of Sublevel Open Stoping (SLOS) and Pillar Retreat methods at a steady-state mining rate of 0.8 Mtpa. The Big Zinc primary mining method is expected to be SLOS, with CRF backfill. The sill pillars are expected to be mined using Pillar Retreat mining method once the adjacent stopes are backfilled.

16.4 Existing Underground Mine Infrastructure

The existing mining infrastructure consists of five surface vertical shafts, and a number of sub-vertical shafts, allowing access to deeper levels. The shafts included in the Kipushi 2017 PFS planning are:

- Shaft 1 (0–650 mRL): Second egress.
- Shaft P1 Bis (400–850 mRL): Second egress.
- Shaft P1 TER (1,138–1,480 mRL): Second egress.
- Shaft 2 (0–500 mRL): Ventilation exhaust.
- Shaft P2 Bis (500–850 mRL): Return ventilation.
- Shaft 3 (0–740 mRL): Second egress.
- Shaft 4 (0–650 mRL): Ventilation exhaust.
- Shaft 4 Bis (650–825 mRL): Return ventilation.
- Shaft 5 (0–1,240 mRL): Personnel, material, services, rock hoisting, and ventilation.
- Shaft P9 (700–1,010 mRL): Second egress.
- Shaft 15 (850–1,172 mRL): Second egress.
- Shaft 19 (825–1,120 mRL): Return ventilation.

The 850 mRL will be utilised as intermediate level on the Shaft 5 to allow personnel and equipment to enter the mine workings, without doing so via the main haulage and crusher level, minimising interactions and downtime to the haulage network.

The main working area is connected to Shaft 5 via the 1,150 mRL main haulage level. There is a crusher chamber at 1,200 mRL; the crusher level is now dewatered. The underground infrastructure exposed since dewatering, is in relatively good order. The crusher is being replaced as the cost of refurbishment was determined to exceed the replacement cost.

A 5 m 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.

A schematic layout of the existing development is shown in Figure 16.20. Digitised shafts, decline and existing development from the 855 mRL to the 1,347 mRL is shown in Figure 16.21.

Figure 16.20 Schematic Section of Kipushi Mine

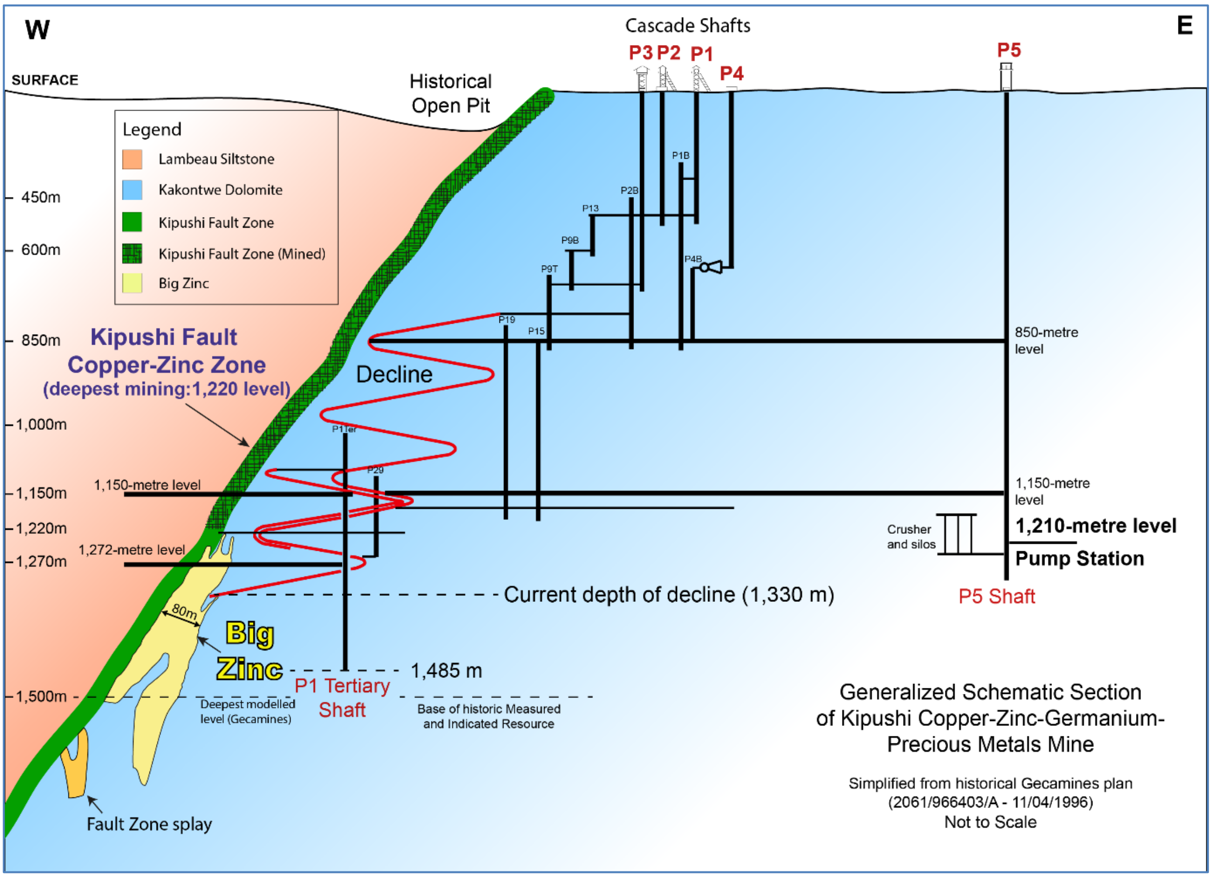


Figure by Ivanhoe, 2016.

Figure 16.21 Digitised Existing Development

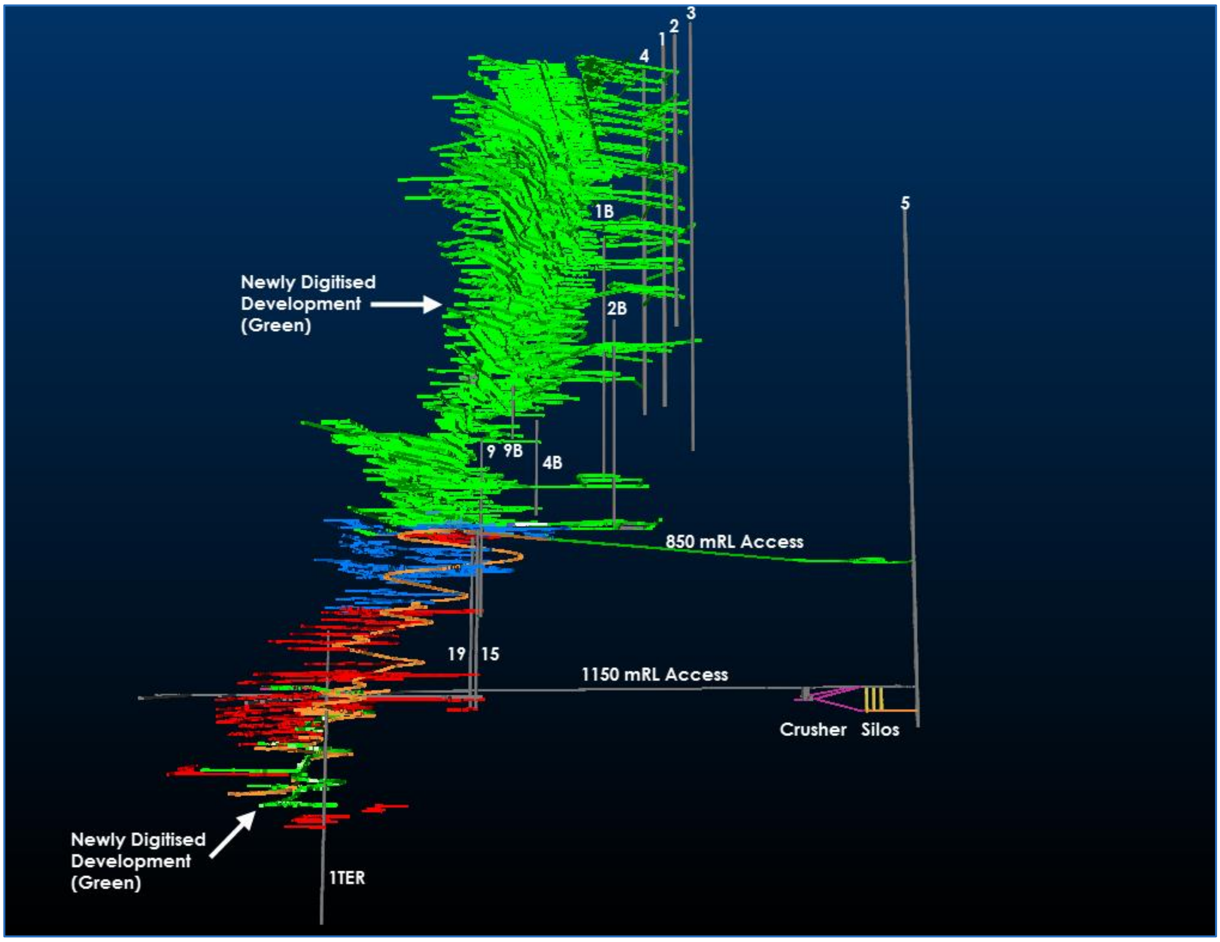


Figure by OreWin, 2017.

A network of underground pumps, cascading dams and pipework currently dewaters the mine at a maximum rate of 3,500 m^3/h. Water is pumped from shafts and sumps to intermediate settling dams on the 1,200 mRL, 1,150 mRL, 1,112 mRL, and 850 mRL levels and then to surface via Shaft 5. The complete existing water handling process flow diagram can be seen in Figure 16.22.

Figure 16.22 Water Handling Process Flow Diagram

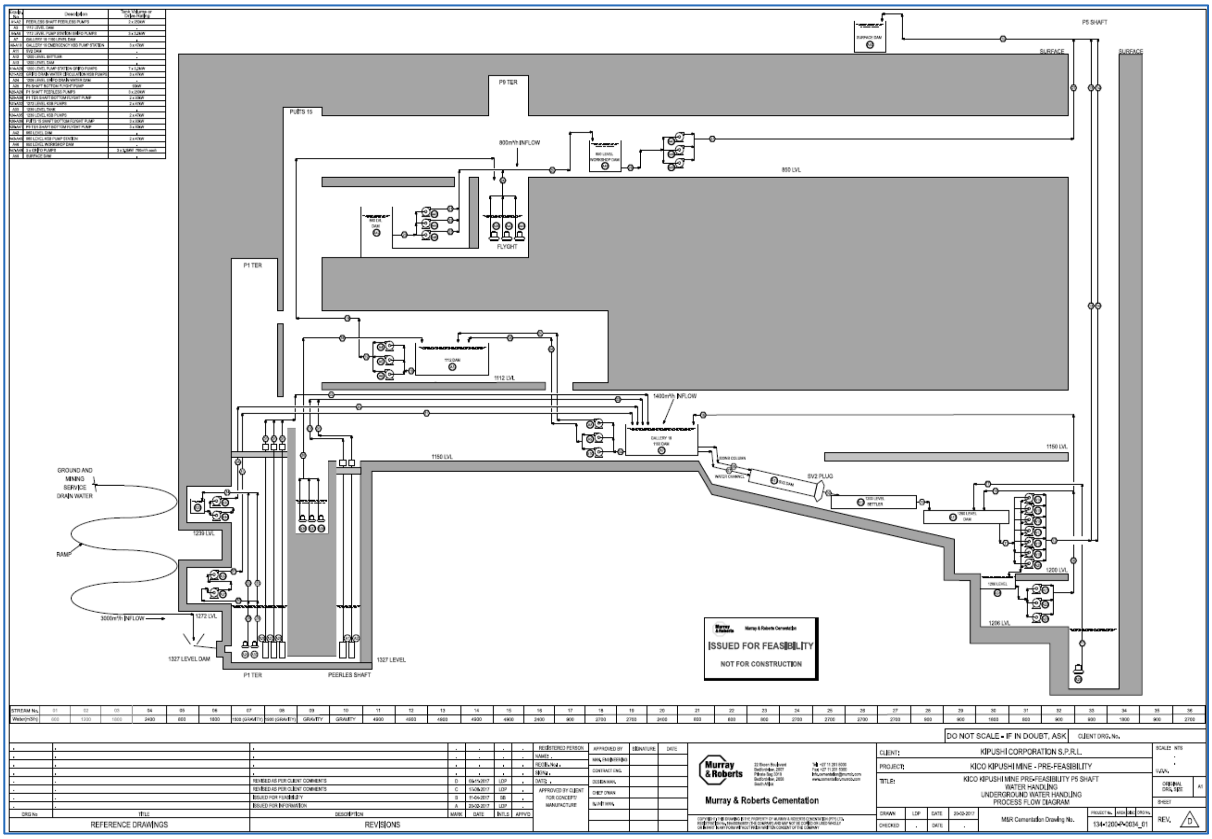


Figure by Murray and Roberts, 2017.

Workshops and magazines exist on the 1,132 mRL and 850 mRL levels. These areas require rehabilitation but will provide locations for machine maintenance, breakdown areas, welding bays, wash bays, tyre changing and storage, explosives storage, lubricant tanks, and diesel storage. Layout of the workshop areas on the 1,132 mRL level is shown in Figure 16.23.

Figure 16.23 1,132 mRL Infrastructure

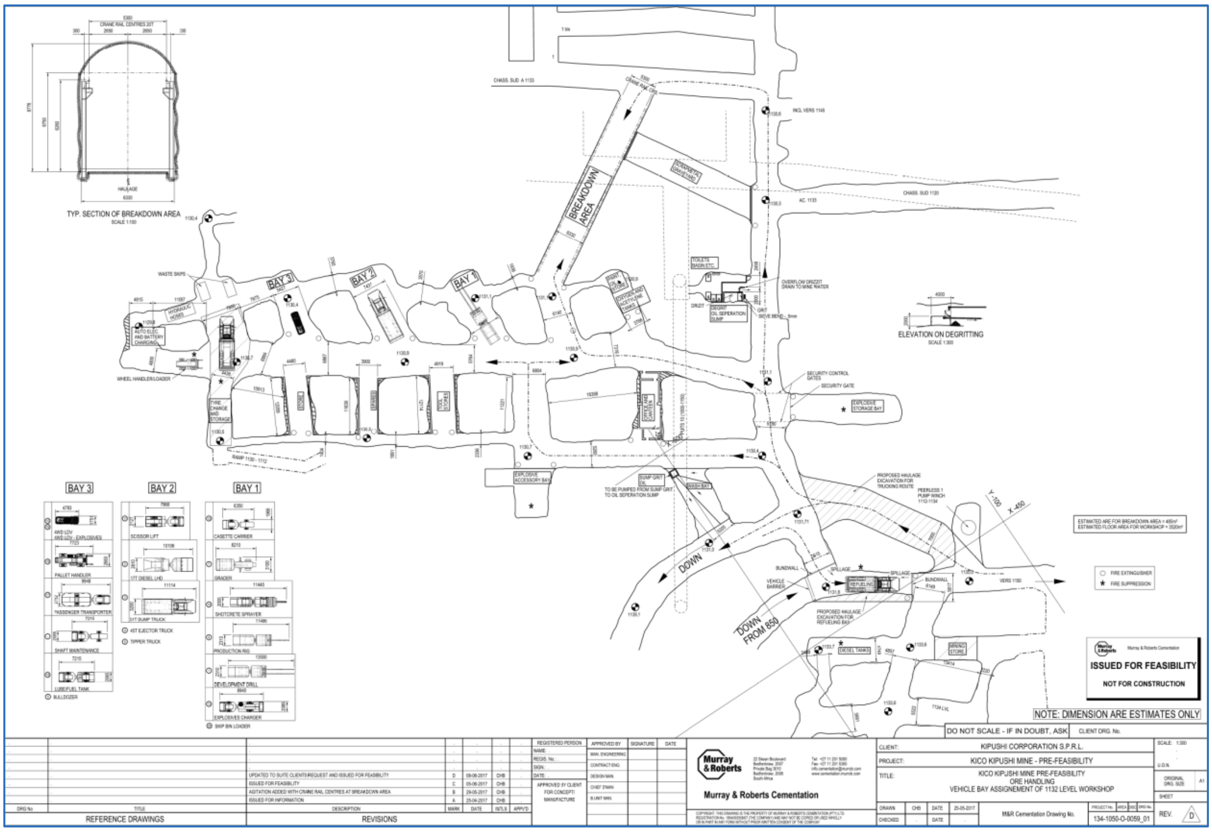


Figure by Murray and Roberts, 2017.

16.5 Mine Design Criteria

16.5.1 Development

Access will be via the existing shafts and internal decline to Big Zinc. The decline will be extended from the current position. Mined material will be trucked to the 1,150 mRL drive crusher tip, fed to the crusher on the 1,200 mRL and then conveyed to silos for temporary storage, before being hoisted to the surface via Shaft 5. The assumptions for Kipushi development are shown in Table 16.28.

Table 16.28 Mine Development Assumptions

Description	Height (m)	Width (m)	Comment
Decline	5.0	5.8	Gradient 1-in-7; Radius 35 m
Lower Lift Extraction Drive	5.0	5.0	–
Drill Drive	5.00	7.00	Second lift and Primary Sill Pillar
Permanent Sill Pillar	6.00	8.00	–
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

Note: As-built width of 855 decline at ~1,300 mRL above Big Zinc is 5.8 m. This needs to be confirmed.

16.5.2 Support

Support classifications for all underground applications are detailed in Table 16.29. Requirements for the decline, stope drives, sill pillar stope drives and access drives are detailed in Table 16.30 and Table 16.31 (SRK (2017) (Kipushi Mine Pre-Feasibility Study Geotechnical Design Parameters_Rev2, pp.10-11).

Table 16.29 Support for Excavations

Support	Area of Application	Support standard
	Shafts (blind sink)	Primary support: Minimum 1.8 m long splits sets at 1.0 m x 1.5 m pattern with mesh. Secondary support: 300 mm concrete lining.
	Vent shafts (Raisebore) (high stress)	Minimum 50 mm shotcrete lining or concrete lining.
S0	Decline. No Fracturing Access drive support for normal conditions (No fracturing) Geological structures	Spot bolting. 2.4 m long tensioned rebar.
S1	Decline. Fracture depth < 0.5 m Access Drive. Fracture depth < 0.5 m	Primary support: 2.4 m long tensioned resin rebar in a 1.8 m x 1.8 m pattern with mesh in the crown down to the midway of the sidewall.
S2	Access Drive (Fracture depth between 0.5 m and 1.5 m) Stope drive support for high stress and dynamic conditions (Fracture depth between 0.5 m and 1.5 m). Stope drive (sill) support for secondary stopes (there will be no re-entry into these stope drives)	3 m long tensioned resin grouted yielding bars in a 1.2 m x 1.2 m pattern with mesh in the crown down to the 0.5 m from the floor.
S3	Stope drive (sill) support for primary stopes as extreme high stress and dynamic conditions are likely (Fracture depth >1.5 m)	3.0 m long tensioned resin yielding bar in a 1.0 m x 1.0 m pattern with 50 mm shotcrete and mesh across the drive, in crown and down to 0.5 m from the floor. Note that shotcrete should be applied first, followed by the installation of bolts and mesh.
	Stope brow support (where necessary)	Primary support + three rows (1.0 m apart) of three 6.5 m long grouted, cable anchors installed within 1.0 m of planned brow position.
	Support of intersections	Primary support + minimum five 4.5 m long pre-tensioned, grouted cable anchors installed in the crown at the time of development.
	Support of large excavations (hoist chambers)	Primary support + pre-tensioned, grouted cable bolts (minimum length = half excavation span), maximum spacing = 0.5 x bolt length.

Table 16.30 Decline and Stope Support Requirements

Excavations	Level From mRL	Level To mRL	Support Required
Decline (long term)	1,200	1,410	S0
	1,410	1,700	S1
Stope Drives (short term)	1,200	1,700	S2
Sill Pillar Stope Drives - Primary (short term)	1,200	1,700	S3
Sill Pillar Stope Drives - Secondary (short term)	1,200	1,700	S2

Table 16.31 Access Drive Support Requirements

Excavations	Level From mRL	Level To mRL	Distance From Stopes		
			15-20 (m)	20-50 (m)	>50 (m)
Access Drives	1,200	1,300	S2	S1	S0
	1,335	1,410	S2	S1	S0
	1,440	1,700	S2	S1	S1

Specifications for support types are shown in Table 16.32 (SRK (2017) (Kipushi Mine Pre-Feasibility Study Geotechnical Design Parameters_Rev2, pp.12).

Table 16.32 Support Specifications

Support Type	Support Specifications
Splits sets (SS 33)	Outer diameter 33.5 mm to 34.2 mm, yield strength 420 MPa black SUPRAFORM steel, minimum steel thickness 2.3 mm, hole size 30 mm to 32 mm.
Rebar	Yield strength 500 MPa black steel, 25–28 mm hole diameter, 20 mm bolt diameter hole size to match rebar diameter for resin mixing (maximum 4 mm annulus or effective mixing demonstrated through approved testing).
Yielding bar	Yield strength 500 MPa black steel, 25–28 mm hole diameter, 20 mm bolt diameter, minimum energy absorption 30 kJ within 300 mm tunnel deformation, hole size to match resin mixing (maximum 4 mm annulus or effective mixing demonstrated through approved testing).
Shotcrete	Minimum 25 MPa shotcrete (28-day strength).
Cable anchor	Minimum 18 mm diameter black steel, 380 kN ultimate load.
Mesh	Black welded mesh, minimum 5 mm gauge, maximum 100 m aperture, blast resistant.
Capsule resin	Two component urethane silicate resin capsules. Fast <30 sec and slow 5 – 10 min setting time.
Injection resin	Two component urethane silicate injection resin with water sealing properties.
Cable grout	Minimum 40 MPa Ordinary Portland Cement, water to cement ratio 0.35 – 0.40.

Based on SRK's geotechnical recommendations, the mine production schedule was used to calculate support requirements and is seen in Table 16.33.

Table 16.33 Support Requirements

	Unit	Decline 5x5.8	Decline 5x5.8	Waste Acc 5x5	Waste Acc 5x5	Waste Acc 5x5	Ore Drive Sill 7x5	Ore Drive Sill 7x5	Ore Drive Sill 8x6	Ore Drive Sill 8x6
Height	m	5.8	5.8	5	5	5	5	5	6	6
Width	m	5	5	5	5	5	7	7	8	8
Area	m^2	29	29	25	25	25	35	35	48	48
Advance per blast	m	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2	3.2
Volume blasted	m^3	92.8	92.8	80	80	80	112	112	153.6	153.6
SG	t/m^3	3	3	3	3	3	3	3	3	3
Tones per blast	t	278.4	278.4	240	240	240	336	336	460.8	460.8
Support Class		S0	S1	S0	S1	S2	S2	S3	S2	S3
Rockbolt length	m	2.4	2.4	2.4	2.4	3	3	3	3	3
Rockbolt diameter	mm	20	20	20	20	20	20	20	20	20
Rockbolt hole diameter	mm	28	28	28	28	28	28	28	28	28
Rockbolt spacing (laterally)	m	1.8	1.8	1.8	1.8	1.2	1.2	1	1.2	1
Rockbolt spacing (horizontally)	m	1.8	1.8	1.8	1.8	1.2	1.2	1	1.2	1
Cable anchor length H/W	m									
Cable anchor diameter H/W	mm									
Cable anchor hole diameter H/W	mm									
Cable anchor spacing H/W (laterally)	m									
Cable anchor spacing H/W (horizontally)	m									
Mesh offset from F/W	m	2.9	2.9	2.5	2.5	0.5	0.5	0.5	0.5	0.5
Shotcrete offset from F/W	m							0.5		0.5
Shotcrete thickness	mm							50		50

Table 16.34 Calculated Support Requirements

Support Requirements	Unit	Total
Total		
Rockbolts 2.4 m	No.	99,400
Rockbolts 3 m	No.	103,652
Anchor bolts	No.	5,828
Mesh	m^2	343,568
Shotcrete	m^3	1,237
Grout	m^3	175
33 mm rock bolt metre drilled	m	549,516
51 mm cable anchor metres drilled	m	29,574

16.5.3 Backfill Strength Recommendations

Numerical elastic modelling was undertaken by SRK on the proposed stope, pillar and development designs, confirming the support requirement recommendations made in Section 16.5.2. Critical failure modes and depth of fracturing determined the stress damage experienced by development drives, ore drives, vertical development and unsupported stopes and pillars. Elastic material properties used for evaluation purposes are shown in Table 16.35.

Table 16.35 Elastic Material Properties

Young's Modulus Gpa	Poisson's Ratio	σ_{h1} (MPa/m)	σ_{h2} (MPa/m)	σ_z (MPa/m)
75	0.3	0.031	0.031	0.0388

It was calculated that some secondary ore drives on the extraction level would require greater support during the extraction of the primary stopes and may require rehabilitation. Ore drives on the sublevel stopes would require greater support during mining of the extraction level stopes. This development is required for backfilling the extraction level and mucking the sublevel and, depending on the depth of fracturing and extent of damage, rehabilitation may be required.

Caving, flexural, sliding, and rotational failure modes were analysed for backfilled stopes based on limit equilibrium criteria in Figure 16.24 (Mitchell, 1999). The critical failure mode for backfilled stopes was determined to be caving which indicated that a CRF strength of 1.2 MPa was required (Shown in Figure 16.24). Secondary stopes with no future exposure require a minimum strength of 400 kPa.

Figure 16.24 Critical Caving Failure Mode

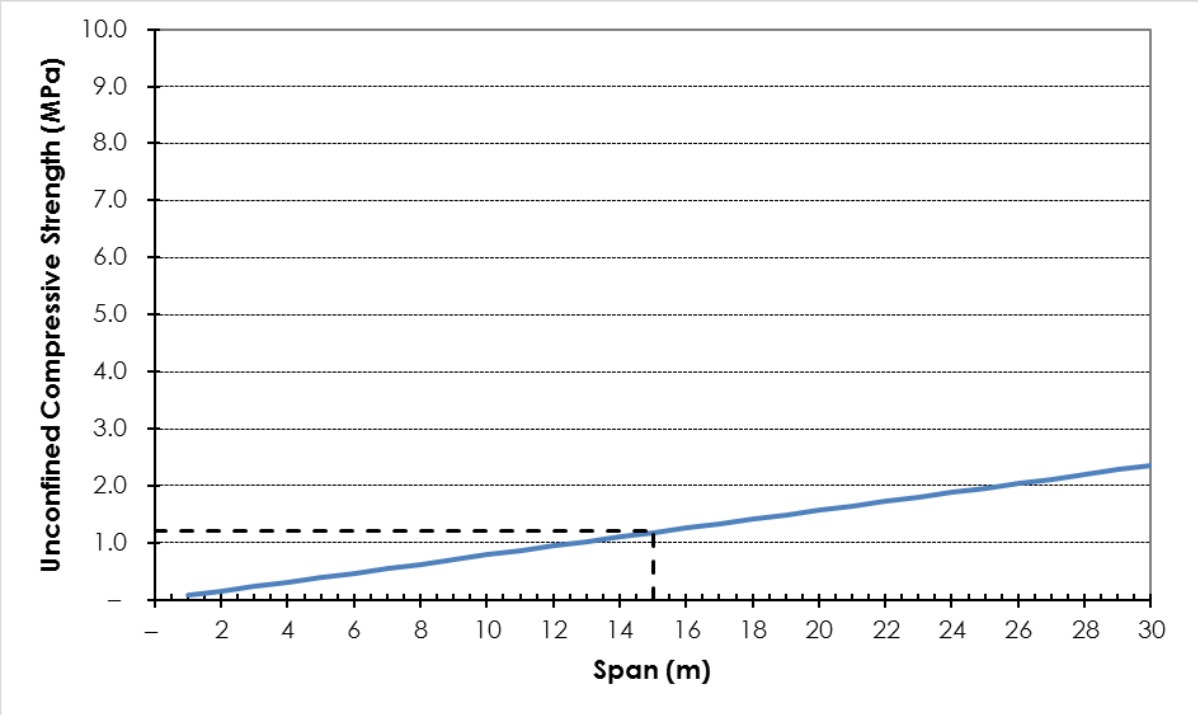


Figure by SRK, 2017.

16.5.4 Loading, Hauling and Backfilling Rates

Once drilled and blasted, material from stopes and pillars will be transported by LHD to either ore or waste stockpiles on each level. From there, material will be trucked to the 1,150 mRL level crusher for hoisting to surface up Shaft 5. Excavated stopes will then be backfilled with CRF, trucked from the CRF plant on the 1,320 mRL level. A section view of the stope backfilling method is shown in Figure 16.25.

Linear haulage distances and travel times for LHD's, ore, waste and backfill trucks, vary dependent on the depth of the active mining level. A level by level TALPAC optimisation was undertaken to calculate the rates, based on machinery numbers for hauling, backfilling and loading. These rates were used for both scheduling and costing purposes.

Figure 16.25 Stope Backfilling Technique

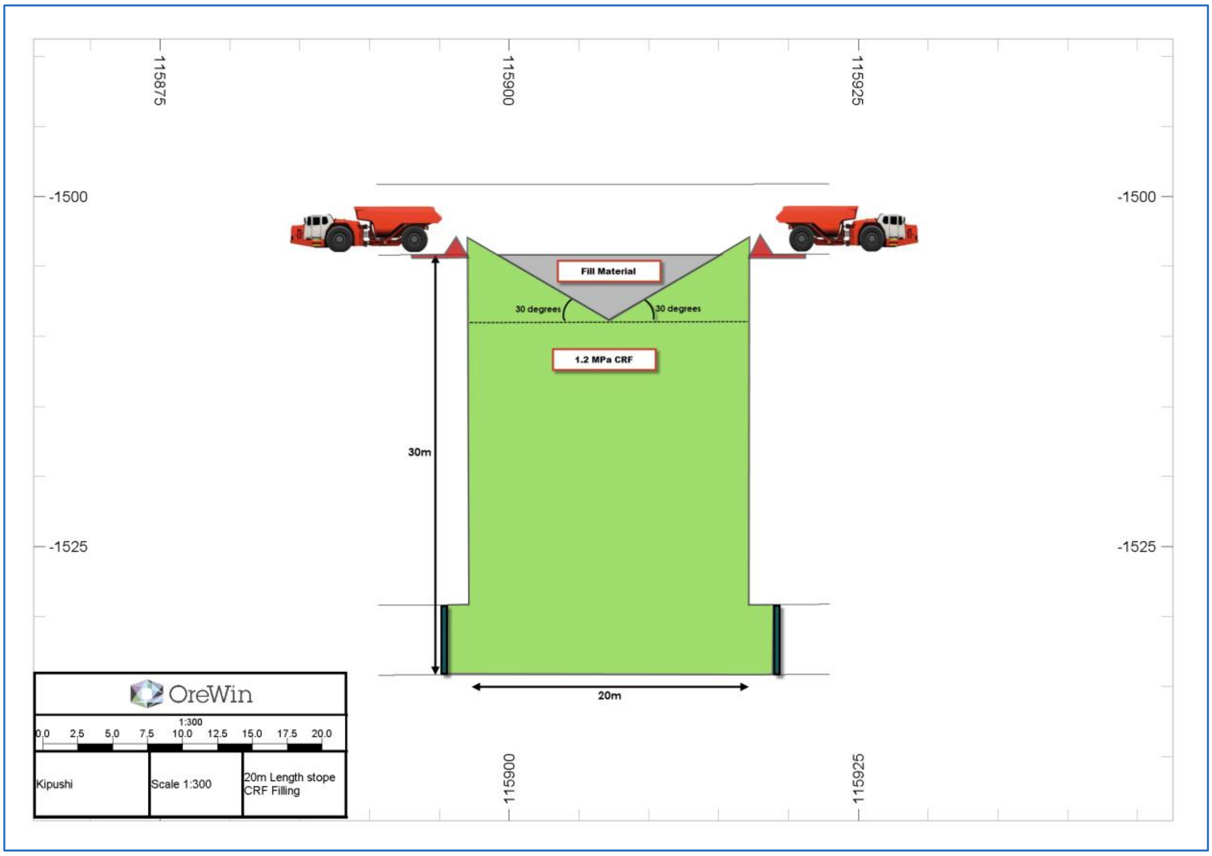


Figure by OreWin, 2017.

16.6 Mine Design

The Big Zinc is located at depths ranging from approximately 1,185–1,710 mRL with the Kipushi 2017 PFS focused on the 1,185–1,590 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 stockpiles and then loaded into trucks. From active mining levels the trucks are expected to haul material to the 1,150 mRL crusher tip.

Figure 16.26 shows all the measured and indicated tonnes above 135 $/t NSR10 cut-off and the outlines of the final stope shapes. Additional blocks lying outside the proposed stope shapes in the Southern Zinc may be included into future stoping shapes with further resource drilling.

Figure 16.26 Measured and Indicated Resource Stopes

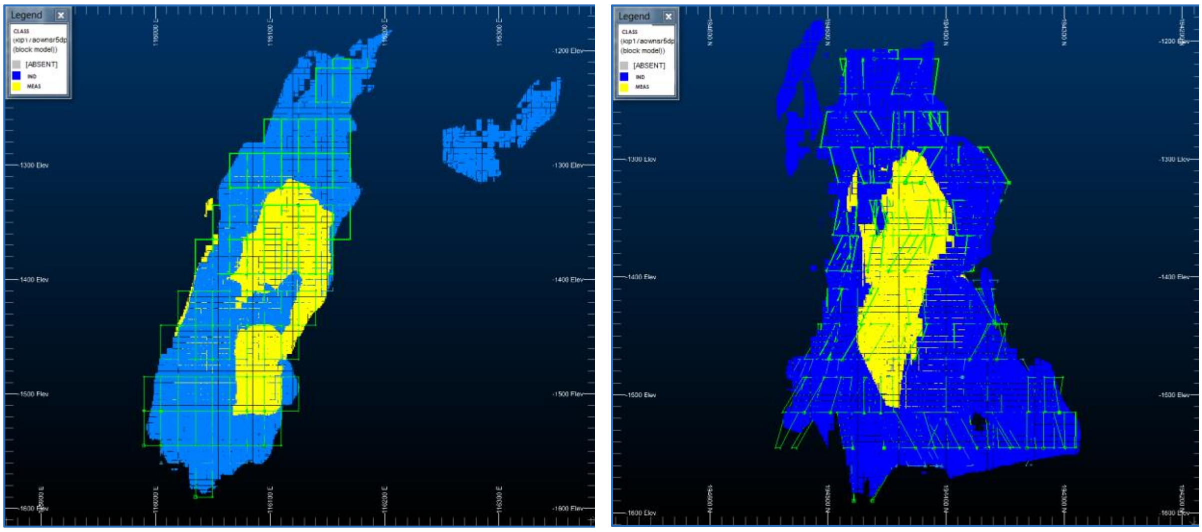


Figure by OreWin, 2017.

Kipushi Big Zinc stopes, existing and planned development and shafts are shown in Figure 16.27 and Figure 16.28.

Figure 16.27 Planned and Existing Development at Kipushi

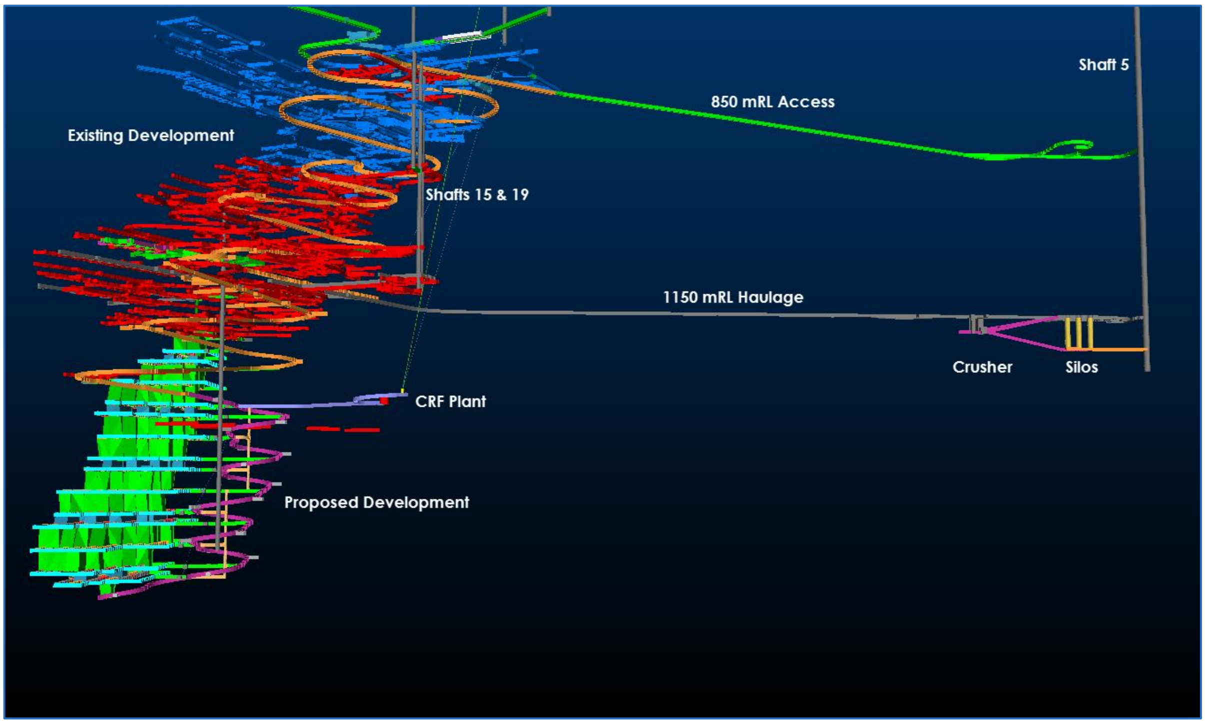


Figure by OreWin, 2017.

Figure 16.28 Planned and Existing Development at Kipushi

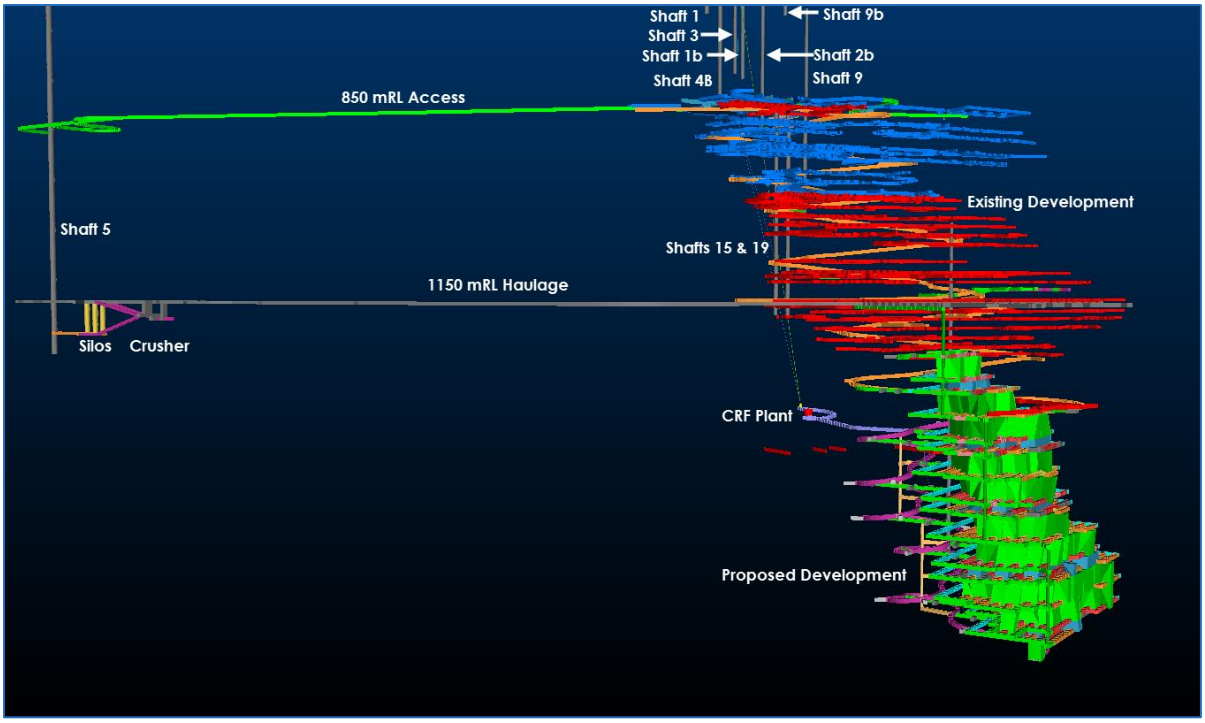


Figure by OreWin, 2017.

16.7 Mining Methods

Mining is planned to be a combination of longitudinal SLOS and Pillar Retreat methods. The Big Zinc mining method is expected to be longitudinal SLOS with mined stopes backfilled with CRF after stoping. Sill pillars are expected to be mined using the Pillar Retreat mining method, once the adjacent stopes are backfilled.

The Big Zinc is expected to be accessed via the existing decline and without significant new development. The decline is planned to be developed from the existing level at approximately 1,330 mRL to the bottom stoping level at 1,590 mRL. The zinc stoping is expected to be carried out between 1,207 mRL and 1,590 mRL, and the uppermost stoping level on the Big Zinc is planned to be the 1,245 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 a single parallel drive in each stope. The sill pillar height is planned to be 15 m. Stopes are planned to be mined 60 m along strike and then filled with CRF. Remote capable loaders are expected to be used for loading the broken rock beyond the stope brow.

16.7.1 SLOS

Longhole stopes are 30 m high which will be separated by 15 m high sill pillars every 60 m and mined with a bottom up mining sequence as seen in Figure 16.29. Stope back and wall support will not be required provided an unfilled stope length of 60 m is not exceeded. The assumptions for SLOS are shown in Table 16.36.

Figure 16.29 Big Zinc Stope Cross-section

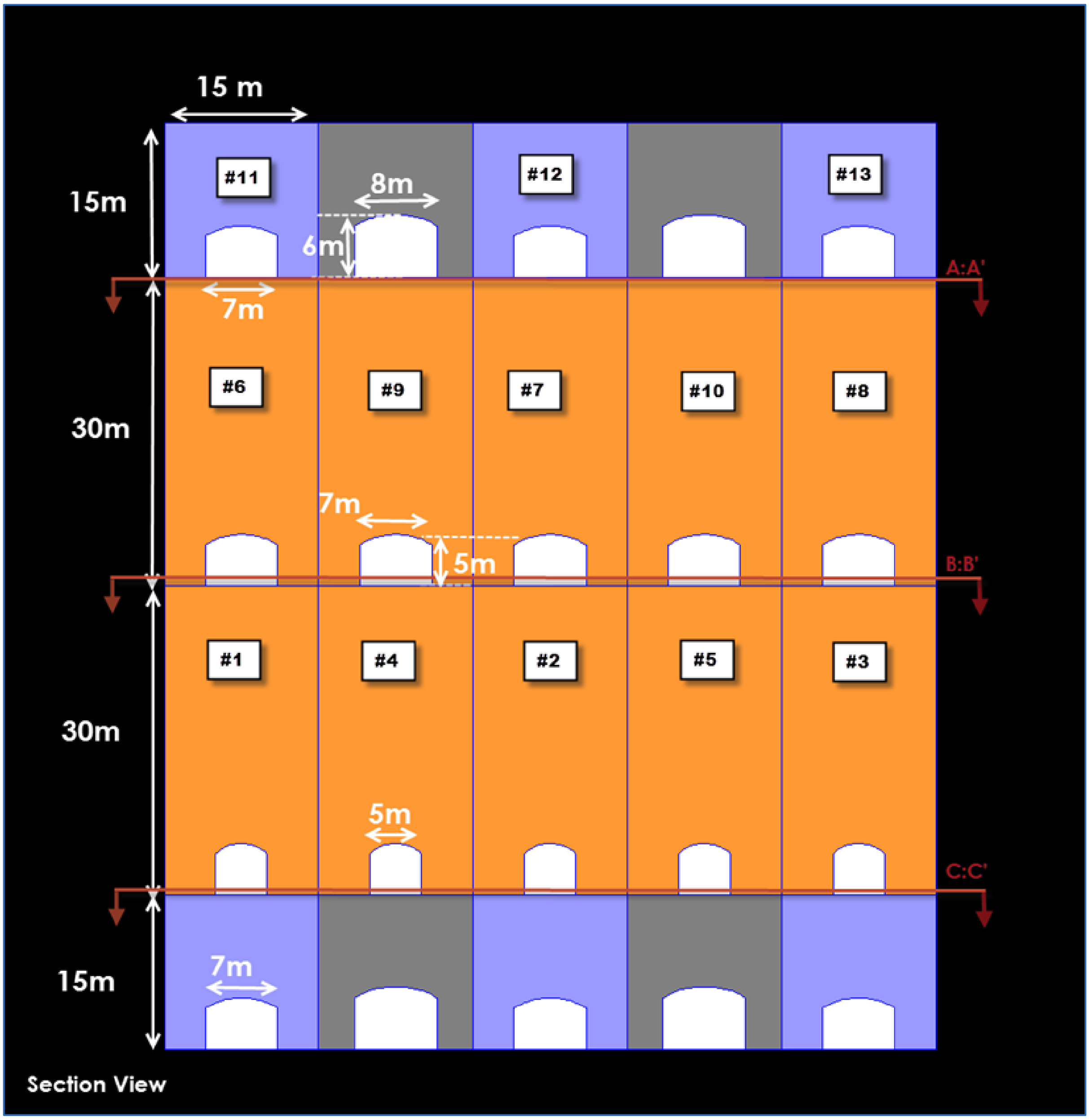


Figure by OreWin, 2017.

Table 16.36 Kipushi Sublevel Open Stoping Parameters

Parameter	Amount/Type
Stoping Direction	North – Longitudinal
Stope Height	30 m
Stope Width	15 m
Stope Length	≤60 m
Stoping Production Rate	Variable by Level
Stope Recovery	90%
Stope Dilution	2.50%
Backfill	CRF (Includes DMS Tailings, and Waste)
Backfill Dilution	2.00%
Maximum Hydraulic Radius - Backs	6 m
Maximum Hydraulic Radius -Walls	10 m

Figure 16.30 and Figure 16.31 respectively show transverse and longitudinal cross-sections of the SLOS method.

Figure 16.30 SLOS Mining Method – Transverse Cross-section

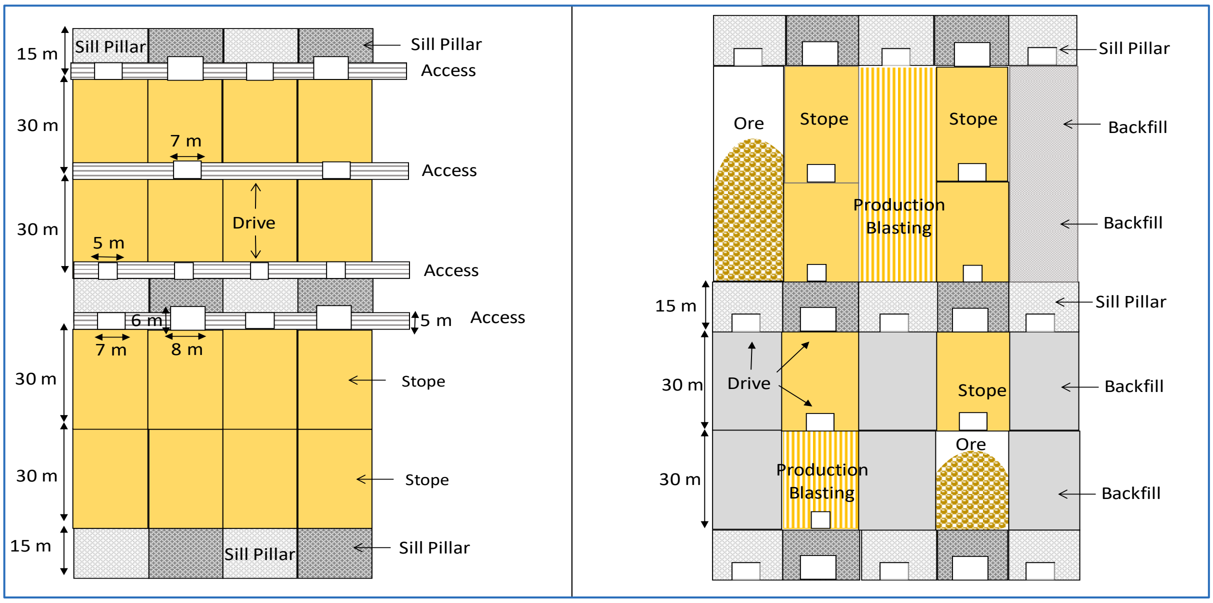


Figure by OreWin, 2017.

Figure 16.31 SLOS Mining Method – Longitudinal Cross-section

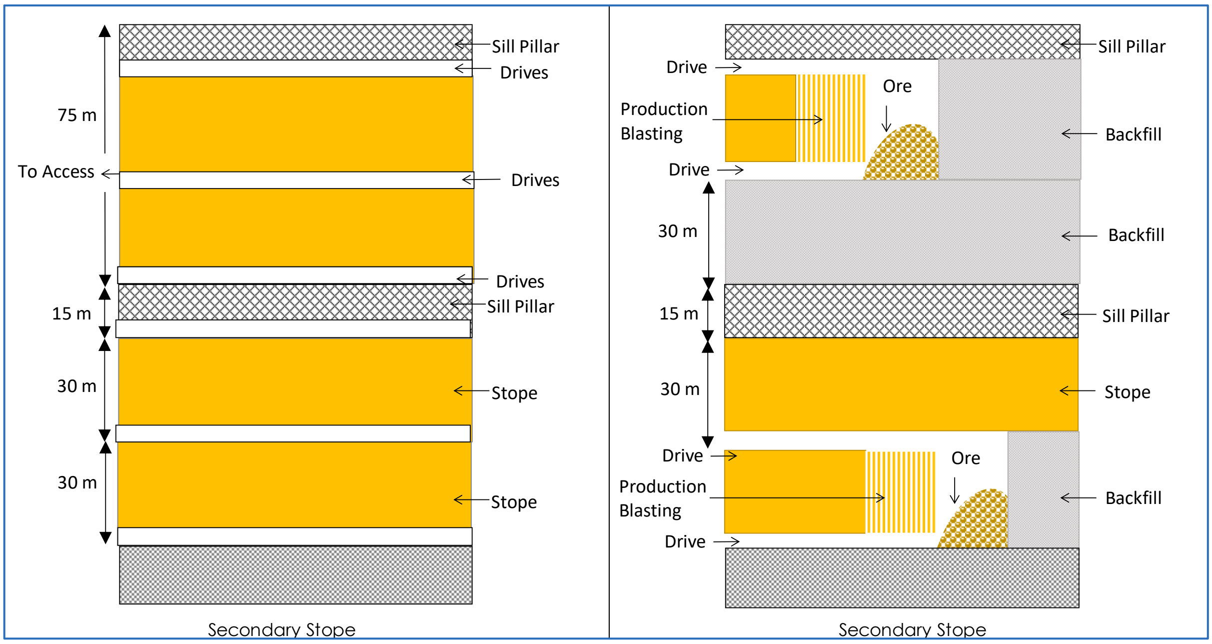


Figure by OreWin, 2017.

Figure 16.32 and Figure 16.33 show the Kipushi zinc stope and development plans at 1,440 mRL and 1,485 mRL respectively. Figure 16.34 shows the Kipushi longitudinal stope and development plan, at the 1,320 mRL extraction Level and the proposed CRF plant location.

Figure 16.32 Kipushi Longitudinal Stope and Development Plan at 1,470 mRL Extraction Level

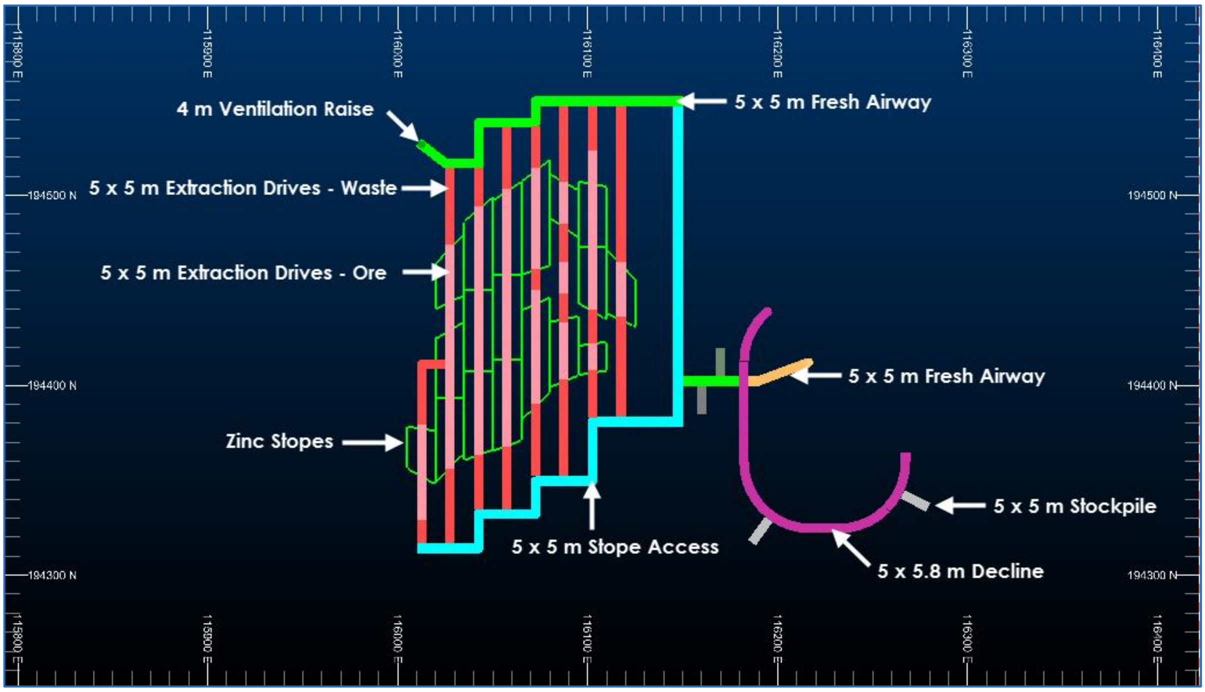


Figure by OreWin, 2017.

Figure 16.33 Kipushi Longitudinal Stope and Development Plan at 1,440 mRL Sublevel

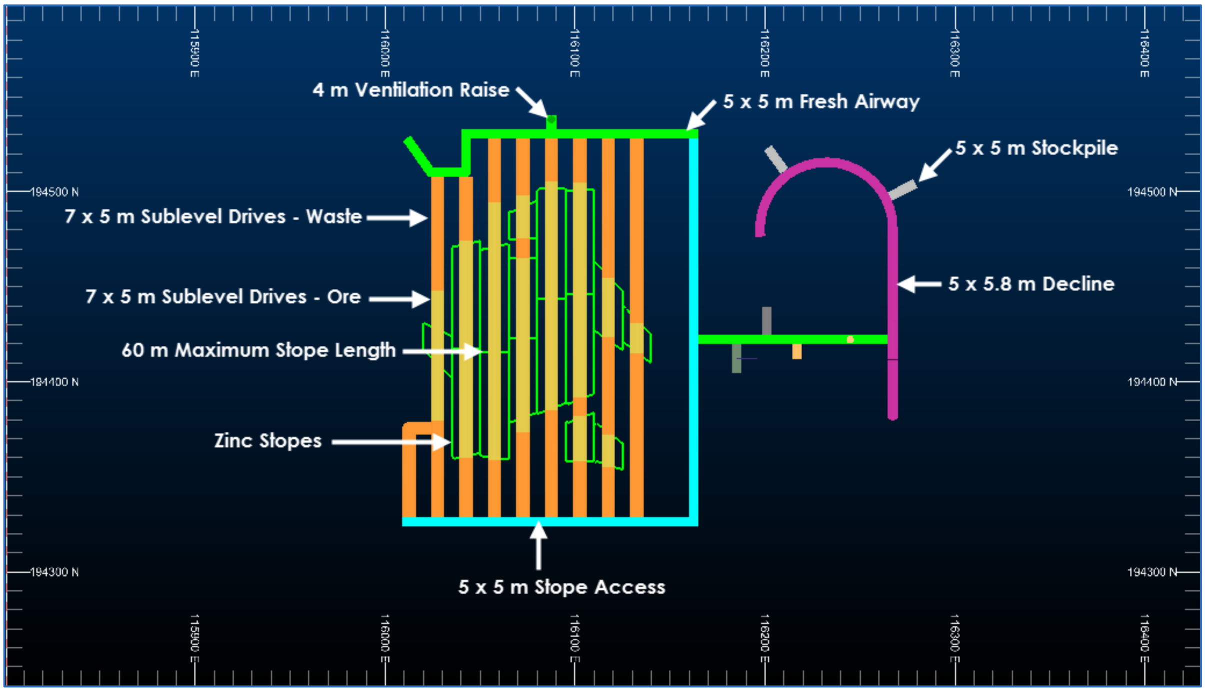


Figure by OreWin, 2017.

Figure 16.34 Kipushi Longitudinal Stope and Development Plan at 1,320 mRL

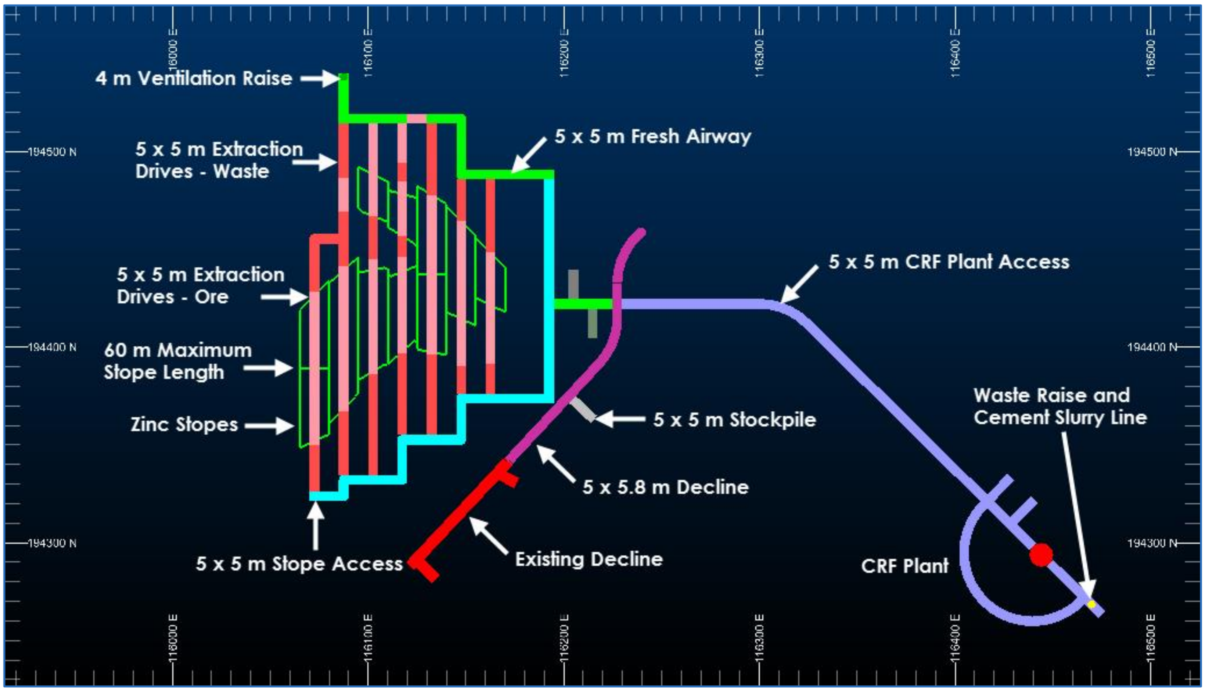


Figure by OreWin, 2017.

16.7.2 Pillar Retreat

The Sill Pillars are 15 m high and occur vertically every 60 m. As seen in Figure 16.30, transversally staggered pillars remain, which contain singular 8 m x 6 m access drives to maximise ore extraction. The extracted sill pillars are not backfilled and are left open for the LOM. Scheduling ensures the pillars are not extracted until the stopes above and below are mined, backfilled and cured. The assumptions for Pillar Retreat are shown in Table 16.37.

Table 16.37 Kipushi Pillar Retreat Parameters

Parameter	Amount/Type
Sill Pillar Height	15 m
Sill Pillar Spacing	60 m
Sill Pillar Production Rate	Variable by Level
Sill Pillar Recovery	59%
Sill Pillar Dilution	20%
Maximum Hydraulic Radius - Walls	6 m

Figure 16.35 shows the Kipushi pillars and development plans on the 1,410 mRL.

Figure 16.35 Kipushi Longitudinal Pillar and Development Plan at 1,410 mRL

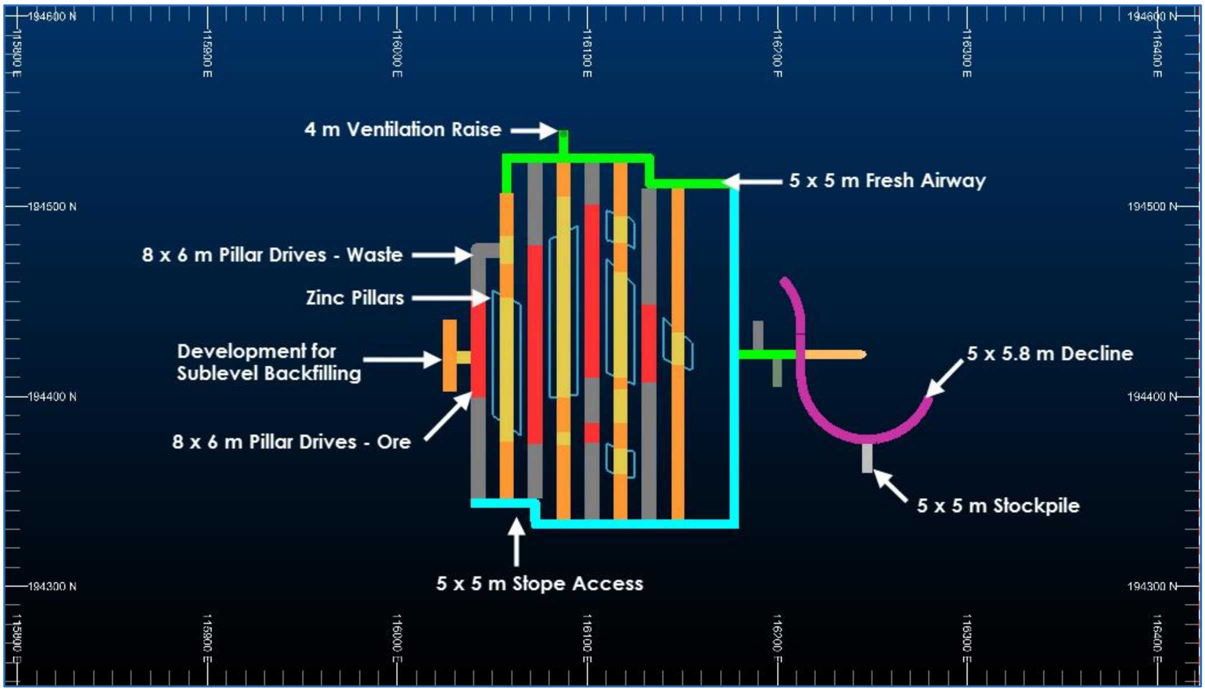


Figure by OreWin, 2017.

16.8 Backfill

CRF of strength 1.2 MPa will be used for stope backfilling. Stockpiled waste rock and DMS tailings will be transported from the surface down a 900 mm borehole to an underground CRF mixing plant on the 1,320 mRL level. A surface loader will feed an aggregate screen and conveyor, which will sort and supply waste material to the waste pass at the required rate.

A surface cement plant will deliver cement slurry down a lined 380 mm borehole to the underground CRF plant. From the surface to the 850 mRL workshop, the borehole will be shared with a diesel line supplying fuel storage tanks in the workshop. From the 850 mRL, another borehole will supply diesel exclusively to the 1,132 mRL workshop for additional storage tanks. From the 850 mRL the lined cement slurry borehole will continue to the 1,320 mRL feeding the CRF plant. Cement will be stored in surface silos of approximately 4 m in diameter which will be sized for one month's capacity.

Cement trucks will supply the silos with regular Portland cement delivered from local cement suppliers. Figure 16.36 shows the process flowsheet for the surface and underground CRF facilities. Both the underground CRF plant and the 850 mRL workshop transfer station will be arranged to allow for water flushing of the lines to reduce the risk of cementing in the pipeline.

Figure 16.36 CRF Process Flowsheet

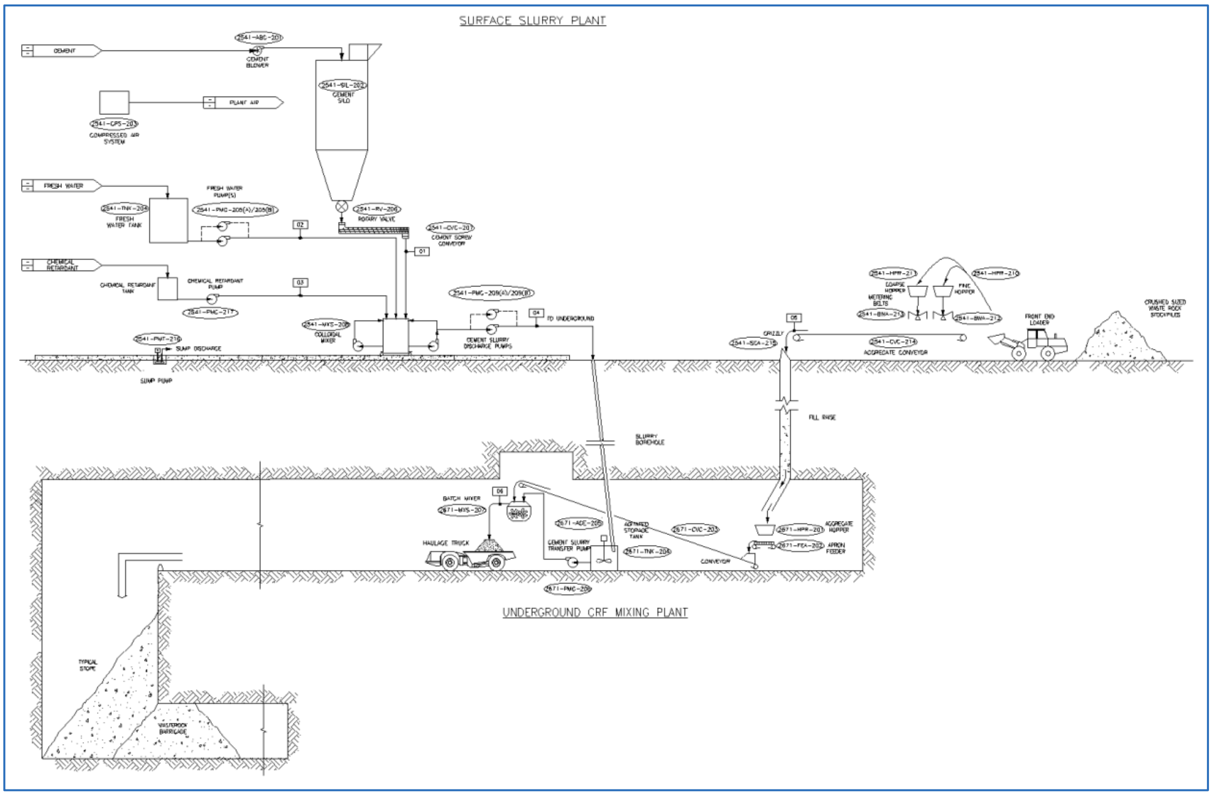


Figure by Golder, 2016.

16.8.1 Composition

The CRF composition depends on strength requirements and available material. Table 16.38 details the testwork on backfill strengths that has been undertaken with varying mixes of underground waste, DMS tails, concentrator fines and cement.

Table 16.38 CRF Strength Backfill Testwork Mix Designs

DMS (%)	Fines (%)	Waste Rock (%)	Cement (%wt)	28 Day Cured Strength (kPa)
20	10	70	5	1,373
20	10	70	7.5	2,716
20	10	70	10	3,572
30	–	70	5	544
30	–	70	10	2,859
–	–	100	5	950

16.8.1.1 Constant Backfill Mix – 100 % DMS Tailings, Remainder Waste

Backfill requirements were added to the process production schedule to determine if there was enough available waste material to supply the backfill demand when 100% DMS tailings was used. As can be seen in Table 16.39, there is a shortfall of 454 kt of waste, which could be sourced from the south end upper benches in the open pit or other on-site sources.

Table 16.39 100% DMS Tailings Used LOM

		Produced	Backfill Required	Backfill Used	Remaining	Additional Required	Maximum Stockpiled
Mined Waste	kt	2,008	2,462	2,008	–	454	718
DMS Tailings (dry)	kt	2,174	1,055	1,055	1,119	–	804
Sub Total	kt	4,181	3,517	3,063	1,119	454	1,522

16.8.2 Surface Waste and DMS Tailings Dump

Surface storage capacity for the DMS tailings and underground waste was limited by the existing surface infrastructure, planned concentrator and DMS plant, conveyors, shafts and the historic open pit. An area was chosen bounded by the road bordering the historic open pit, the proposed concentrator, DMS plant sites and roads within the mining lease.

This footprint ensures load and haul distance for feeding the waste pass was minimised and the proximity to the waste conveyor from Shaft 5 and the DMS tailings conveyor from the DMS plant was maintained. Depending on the degree of mineralisation of the DMS tailings and waste, the dump base may need to be constructed with a protective cap of unmineralised material, to prevent environmentally harmful mine run-off.

Due to the closeness of the surface mine infrastructure and proposed dump to the local community, the northern face was designed with future rehabilitation in mind. Table 16.40 shows the dump design criteria.

Table 16.40 Waste and DMS Tailings Dump Design Criteria

Northern Face - Parameters		
20.0	degree	Overall face angle
29.8	degree	Bench Face Angle
5.0	m	Bench Height
5.0	m	Berm Width
All Other Faces - Parameters		
23.5	degree	Overall face angle
55.0	degree	Bench Face Angle
5.00	m	Bench Height
8.00	m	Berm Width

Volume calculations show there is sufficient storage on surface to cater for the waste rock from initial underground development, as well as the DMS tails produced before backfilling commences. Figure 16.37 shows the 1.5 Mt waste dump design in Year 6 for the backfill mix of 100% DMS tailings and the remainder being waste.

Figure 16.37 Case: DMS Tailings 100% Used LOM - Max Waste and DMS Tailings Stockpile

Figure by OreWin, 2017.

16.9 Fuel Transfer System

The maximum underground diesel fuel storage capacity cannot exceed one week's work requirement and the maximum combined capacity of the surface batch tank and delivery pipeline must be less than 50% of the largest underground storage tank (DMIRSWA (1997). Diesel Transport, Storage and Refuelling Underground Guideline, pp.4-6). The size of the underground storage tank is based on equipment fuel requirements.

Fuel is supplied via a 1,325 m long fuel line, from the surface to the 850 mRL workshop and from the 850 mRL workshop to the 1,332 mRL workshop. Underground storage tanks are fed via a batching system where the surface supplies the underground as required. The batch tanks are fed from surface storage tanks or directly from a fuel tanker.

Four storage tanks of 18 kL capacity each will store fuel underground. The maximum combined capacity of the batching system and piping therefore cannot exceed 9 kL.

Fuel will be piped to hose reel stations for underground equipment refuelling. The fuelling station will have the storage tanks and pumps installed in an enclosed drift with fire doors and appropriate fire suppression systems. Location of the underground fuel storage facility on the ventilation circuit exhaust side will ensure in the event of a fire, fumes and smoke do not enter the operational part of the mine.

16.10 Mine Equipment Requirements

16.10.1 Equipment Criteria

Criteria considered in equipment selection included suitability, equipment standardisation, existing equipment and cost. The equipment selection process was iterative and aimed at obtaining the optimum equipment required to achieve the planned development and production quantities and rates. The TALPAC haulage optimisation software package was used to optimise the sizes and quantities of LHD's and trucks and calculated a variable mining rate by level.

The equipment requirements for the Kipushi project are split into two categories, fixed equipment and mobile equipment. The equipment requirements for each category cover the major components for the operation. The following are the design criteria for sizing, selecting, and quantifying fixed and mobile equipment.

- Mining Method.
- Mined Ore Production Rate 0.8 Mtpa.
- Ventilation Requirements.
- Mine Design Criteria.

Costs for mobile and fixed equipment are based on the following criteria.

- Truck and loader quotes and specifications from Sandvik.
- Fan quotes from Howden.
- Pump quotes from Weir and Sulzer.
- Contractor labour, mining and equipment rates from Byrnecut.
- Mobile equipment quantities, purchases, and rebuild schedules are per the Kipushi LOM plan.

16.10.2 Mobile Equipment

The mobile equipment required for lateral development includes drill jumbos, LHDs, haul trucks, and ground support equipment. Mobile equipment required for stoping includes longhole drill rigs, LHDs, haul trucks, and ground support equipment. The key underground mobile mining equipment includes:

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

The required and planned numbers of all mobile equipment are shown in Table 16.41, and Figure 16.38 to Figure 16.41.

Table 16.41 Mobile Equipment in Service

	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
	-1	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**
Development Drill	1	1	2	2	2	2	2	2	2	1	1	1	1
17 t Diesel LHD	1	2	2	3	3	4	4	4	3	3	3	3	3
51 t Dump Truck	1	2	4	6	6	6	7	6	6	5	5	5	5
Support Bolting	1	1	1	1	1	1	1	1	1	1	1	1	1
Production Drill	–	2	2	2	2	2	2	2	2	2	2	2	2
Scissor Lift	1	1	1	1	1	1	1	1	1	1	1	1	1
Underground Grader	1	1	1	1	1	1	1	1	1	1	1	1	1
UG Cassette Carrier	1	1	1	1	1	1	1	1	1	1	1	1	1
Explosive Charger	2	2	2	2	2	2	2	2	2	2	2	2	2
4WD LDV - Explosives	1	1	1	1	1	1	1	1	1	1	1	1	1
4WD LDV	8	8	8	8	6	6	6	6	6	6	6	6	6
Passenger Transporter	2	3	3	3	3	3	3	3	3	3	3	3	3
Lube / Fuel Truck	1	1	1	1	1	1	1	1	1	1	1	1	1
Pallet Handler	1	1	1	1	1	1	1	1	1	1	1	1	1
Skip Bin Loader	1	1	1	1	1	1	1	1	1	1	1	1	1
Tipper Truck	1	1	1	1	1	1	1	1	1	–	–	–	–
Wheel Handler	1	1	1	1	1	1	1	1	1	1	1	1	1

Figure 16.38 Mobile Equipment – Development Drills

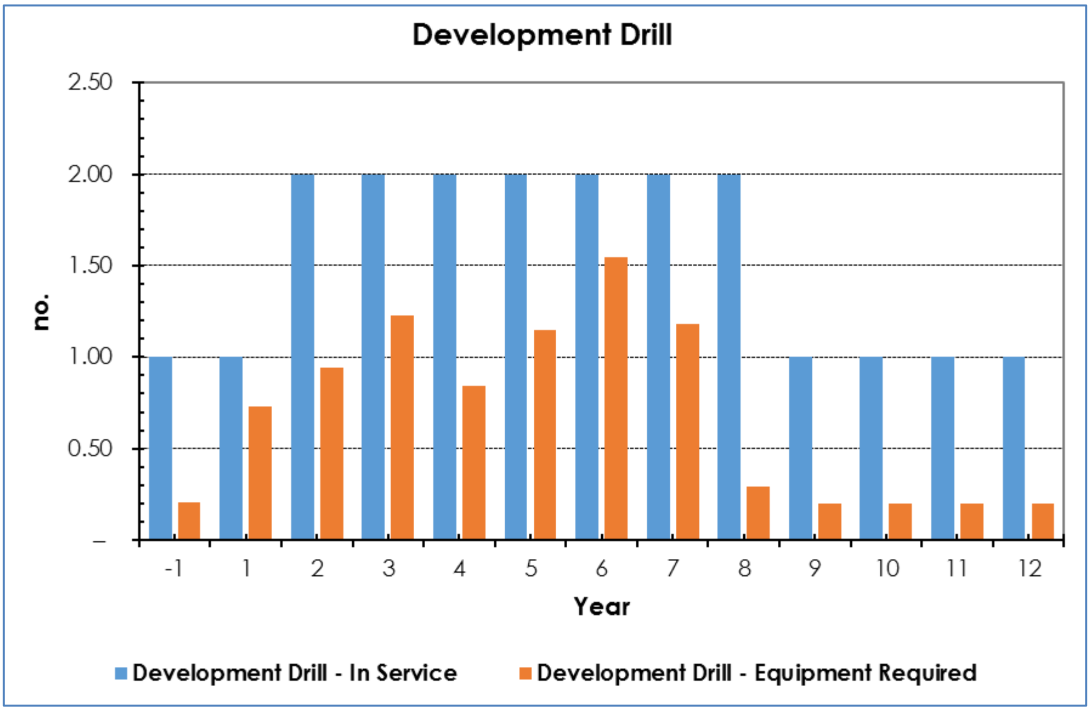


Figure by OreWin, 2017.

Figure 16.39 Mobile Equipment – Loaders

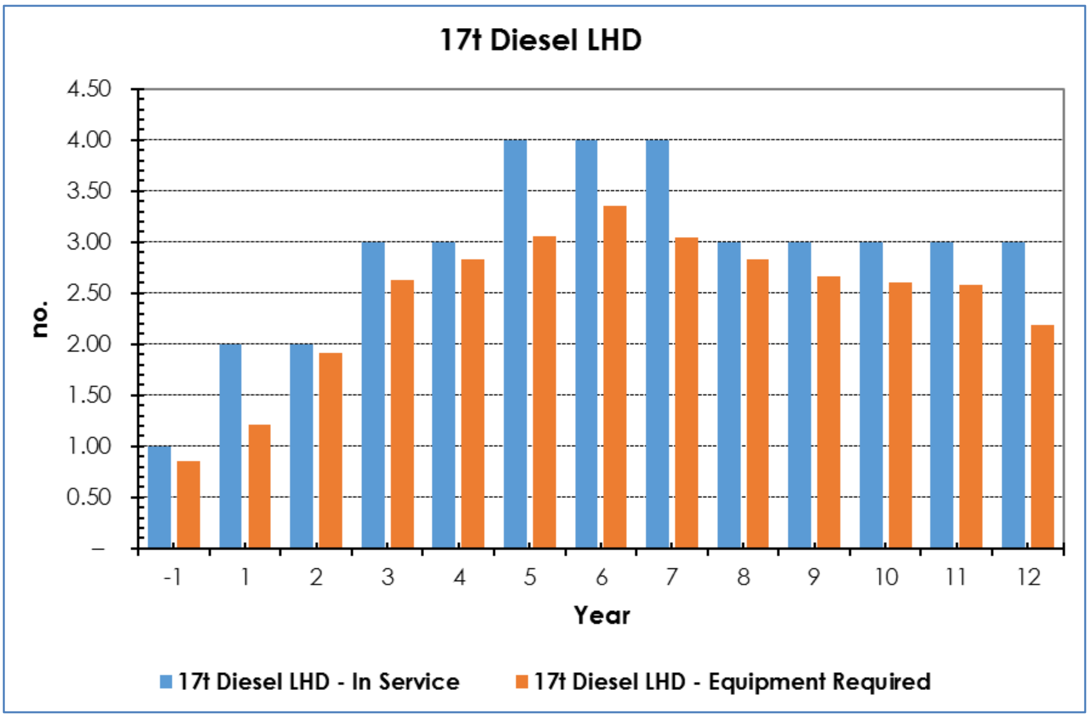


Figure by OreWin, 2017.

Figure 16.40 Mobile Equipment – Production Drills

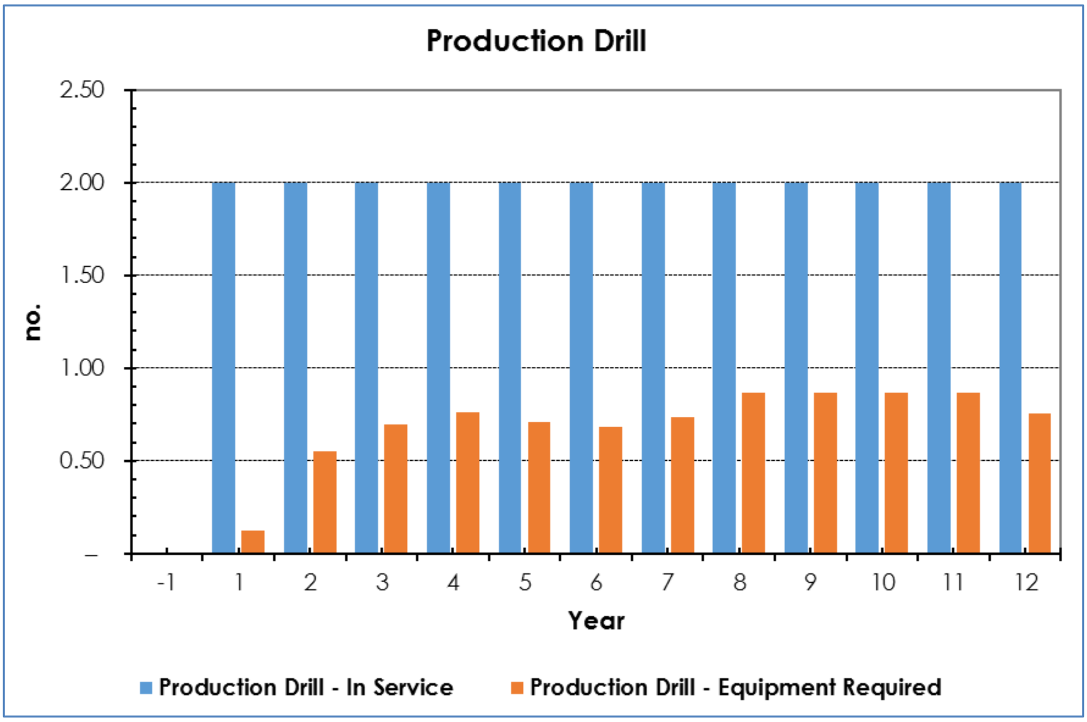


Figure by OreWin, 2017.

Figure 16.41 Mobile Equipment – Trucks

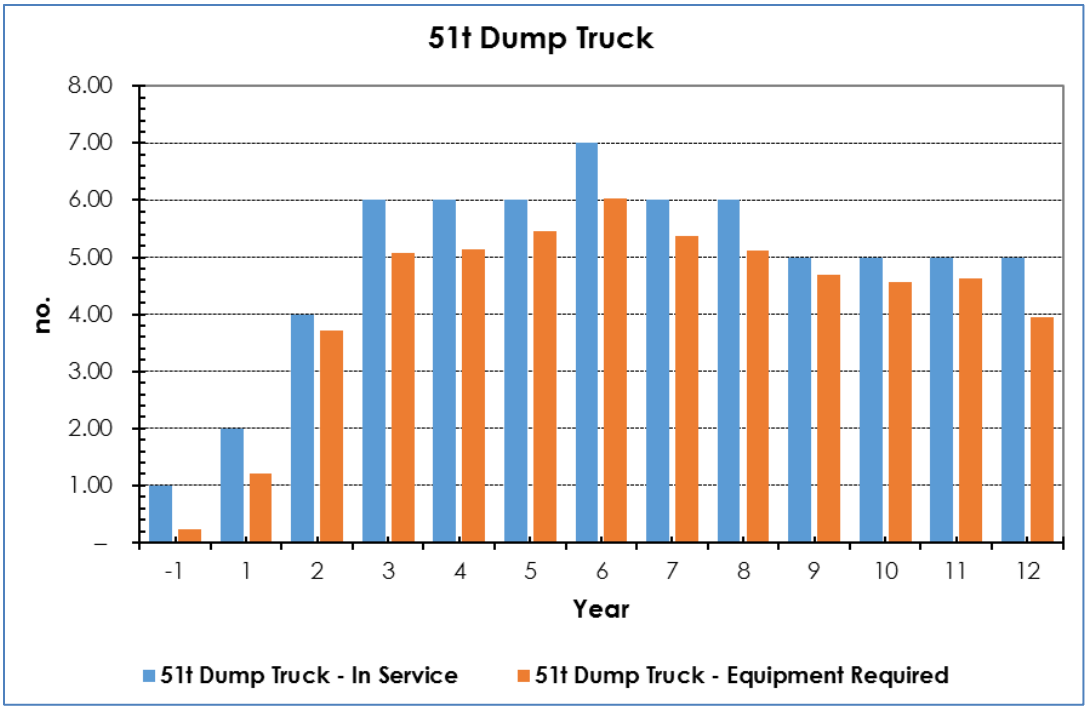


Figure by OreWin, 2017.

16.10.3 Fixed Equipment

Due to the historic nature of Kipushi and the fact it is currently under care and maintenance, significant underground fixed equipment exists in place. Existing underground infrastructure is also detailed in Section 16.4 and includes but is not limited to:

- Shaft Winders.
- Skips and Cages.
- 850 mRL and 1,135 mRL Workshop Facilities.
- 1,150 mRL Silos.
- 1,150 mRL Conveyor.
- Dewatering Pumping Infrastructure.

16.11 Personnel

The site personnel are provided partially by the client and partially by the contractor. Both provide a combination of expatriates and nationals. The expatriates are employed at the beginning of the project, to be replaced by nationals as the project goes on. The client provides labour for roles from the surface down to and including the crusher while the contractor provides labour from the crusher down to the face.

KICO will provide labour for all the roles to the crusher. This includes the main pumping stations, crusher, levels, winders and surface operations. KICO have supporting roles such as the General Manager, supervisors, foremen, and maintenance staff. KICO provide majority of the technical staff such as engineers, surveyors and geologists. At the start of the project various roles are filled by expatriate employees, but as the project progresses these roles are filled by the local workforce. The General Manager however remains an expatriate for the whole project. The contract labour is made up of all personnel from the face to the crusher. This includes all the drilling and blasting, material excavation and transport to crusher. The contractor also provides supporting technical and management roles such as supervisors, managers, maintenance staff and safety. At the start of the project various roles are filled by expatriate employees, but as the project progresses these roles are filled by the national workforce.

16.12 Mine Development Plan and Schedule

Critical activities in developing the underground mine and ramping up to full production include the following items.

- Completion of CRF plant development.
- Completion of the waste raise and cement slurry line to allow for stope backfilling.
- Developing access to drill stopes in the upper portion of the Big Zinc.
- Development and construction of critical surface and underground infrastructure.
- Development of critical access development and ventilation airways.

The mine schedule is based on decline development beginning on Q4 2018 from the 1,330 mRL level existing decline. Pilot hole drilling from the surface for the waste raise and cement slurry line can only begin once the underground CRF plant development is complete. In addition to this critical path, development to the upper stopes in the Big Zinc will importantly yield ore tonnes that can feed the processing plant and bring forward its construction.

16.12.1 Development Productivity Rates

Development rates were calculated from first principles and the SRK supplied ground support requirements. Table 16.42 lists the lateral development rates that were used in the EPS schedule. Development crews drive multiple headings whenever possible, and by doing so, increase utilisation of crews and equipment.

Table 16.42 Lateral Development Rates

Development Type	Rate
Decline (first month)	80 m/month
Decline (until waste raise reached)	165 m/month
Decline (once waste raise reached)	80 m/month
CRF plant/waste raise access	165 m/month
Waste access to stopes	80 m/month
Fresh airways	80 m/month
Sumps and stockpiles	80 m/month
5 m x 5 m stope development drives	66 m/month
7 m x 5 m stope development drives	66 m/month
8 m x 6 m stope development drives	66 m/month

All internal ventilation raises and ore passes are designed to be raisebored. All raiseboring assumes that the drill rigs, drill pipe, bits, reaming heads, and crews are on site and available as necessary. Vertical advance rates exclude mobilisation and demobilisation of the raiseboring rig and crews. Advance rates are applied in accordance with raise diameter and length. Table 16.43 lists the lateral development rates that were used in the EPS schedule.

Table 16.43 Vertical Development Rates

Development Type (Vertical)	Rate
Vent Raise	3 m/day
Waste Raise	3 m/day
380 mm pilot hole for waste raise	50 m/day
380 mm pilot hole for cement/diesel line	50 m/day

16.12.2 Development Quantities

Development quantities for LOM lateral and vertical development are shown in Table 16.44, Figure 16.42, and Figure 16.43.

Table 16.44 LOM Development Quantities

	Units	Total	Schedule Year												
			-1	1	2	3	4	5	6	7	8	9	10	11	12
Lateral Development															
Zinc Decline (m)	m	2,523	167	493	177	398	418	524	345	–	–	–	–	–	–
Zinc Access (m)	m	6,086	271	885	1,409	575	837	1,073	1,034	–	–	–	–	–	–
Zinc Drive 5x5 (m)	m	2,827	–	404	539	487	105	655	102	488	47	–	–	–	–
Zinc Drive 7x5 (m)	m	4,068	–	218	354	709	524	729	1,038	496	–	–	–	–	–
Zinc Drive 8x6 (m)	m	1,309	–	–	163	284	209	93	319	241	–	–	–	–	–
Waste Drive 5x5 (m)	m	3,123	–	387	443	503	139	468	248	751	185	–	–	–	–
Waste Drive 7x5 (m)	m	4,841	69	341	398	708	786	816	1,036	688	–	–	–	–	–
Waste Drive 8x6 (m)	m	1,635	–	–	107	289	280	130	345	435	49	–	–	–	–
Fresh Air Way 5x5 (m)	m	2,557	160	406	412	373	195	275	735	–	–	–	–	–	–
Waste Pass Access 5x5 (m)	m	437	90	347	–	–	–	–	–	–	–	–	–	–	–
Total Lateral Development	m	29,405	757	3,481	4,004	4,326	3,493	4,763	5,201	3,098	281	–	–	–	–
Vertical Development															
Ventilation Raise 4 m (m)	m	645	–	180	60	120	75	105	105	–	–	–	–	–	–
Pump Line Development 6" (m)	m	204	–	–	61	12	58	35	37	–	–	–	–	–	–
CRF Vertical (m)	m	19	–	19	–	–	–	–	–	–	–	–	–	–	–
Cement/Diesel/Waste Pilot 380 mm (m)	m	2,976	1,187	1,790	–	–	–	–	–	–	–	–	–	–	–
Waste Raise 900 mm (m)	m	1,325	–	626	699	–	–	–	–	–	–	–	–	–	–
Total Vertical Development (m)	m	5,170	1,187	2,615	821	132	133	140	142	–	–	–	–	–	–
Production Drilling	m														
Stope+Pillar Production Drilled (m)	m	550,205	–	7,924	61,110	39,271	49,903	65,507	53,120	39,431	64,348	44,322	52,158	53,939	19,172

Figure 16.42 Total Lateral Development Meters

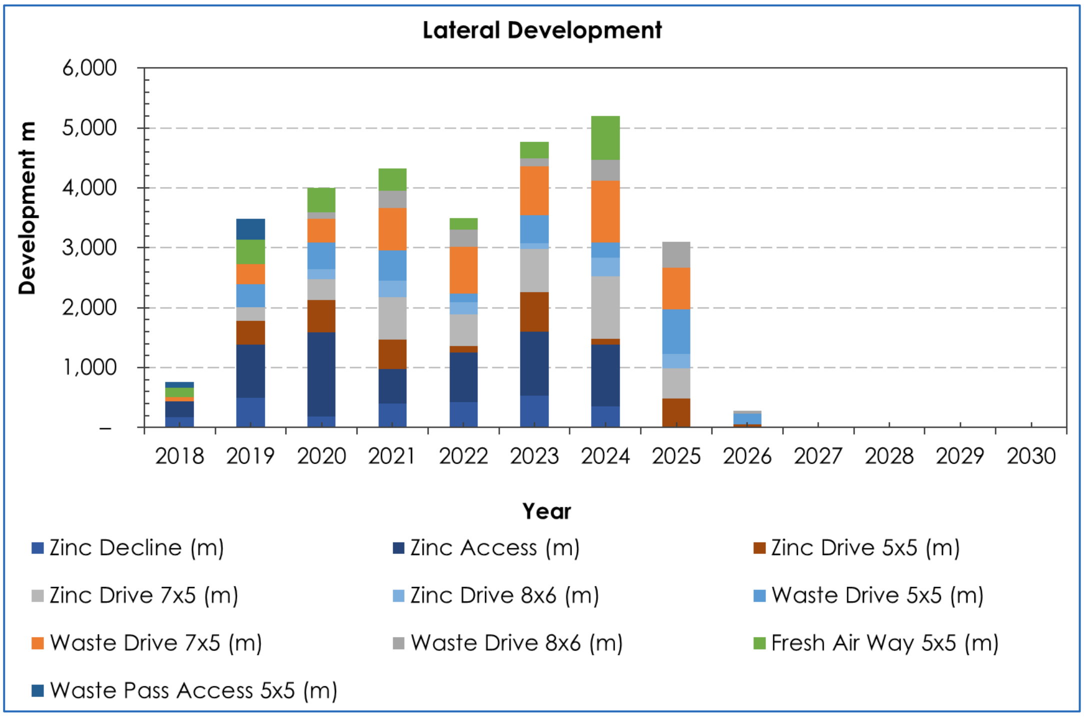


Figure by OreWin, 2017.

Figure 16.43 Total Vertical Development Meters

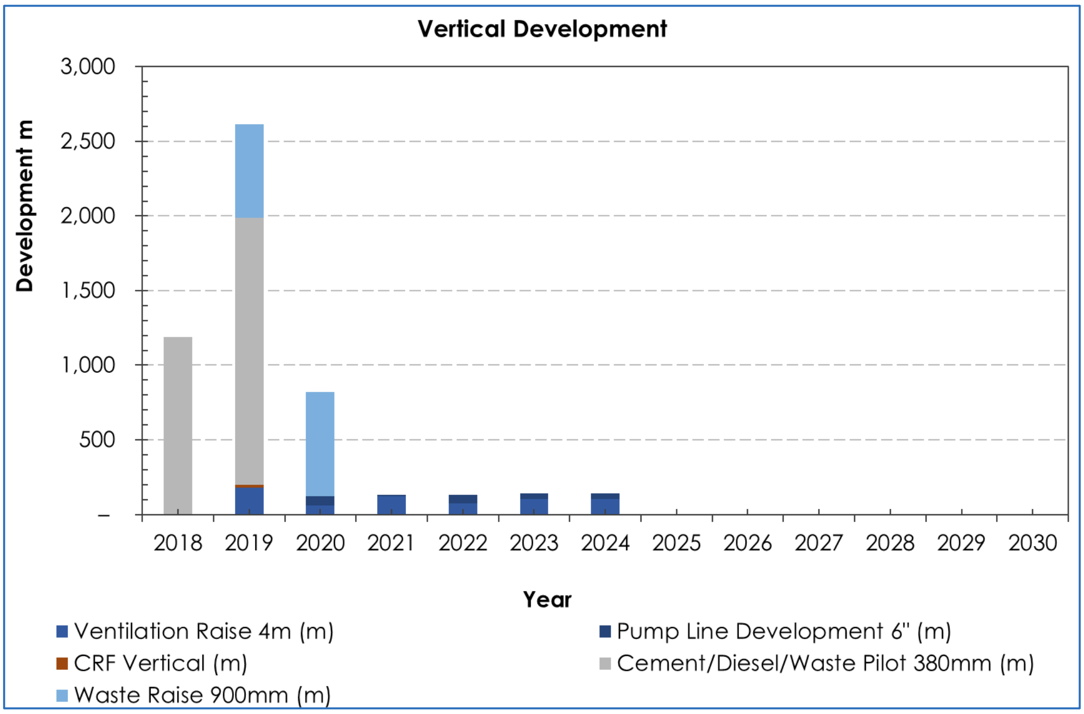


Figure by OreWin, 2017.

16.13 Ore and Waste Handling System

- On each mining level material (waste+ore) is loaded with LHD's to stockpiles.
- Material is then loaded into trucks, which transport material up decline to the 1,150 mRL level.
- Trucks dump into an 800 mm x 800 mm Grizzly.
- Ore and Waste Bin Capacity.
- Ore from the bins enters the plate feeder (Sandvik SH1351M).
- Rock less than 200 mm is diverted directly to the hoisting ore bins.
- Rock Greater than 200 mm to the Jaw Crusher.
- Crushed material enters the plate feeder (Sandvik SP1426).
- Ore conveyed to the crushed ore storage silos. Shown in Figure 16.45.
- All material is then hoisted in skips up Shaft 5.
- Surface conveyors transport ore to the DMS plant.
- Surface conveyors transport DMS tails to surface stockpiles.
- Surface conveyors transport DMS concentrate to concentrator.
- Piping pumps DMS tails to TSF or cement slurry plant.
- Concentrate is transported in 2 t bags to the train load out facility.
- Surface conveyors transport waste to surface stockpiles.
- Waste/DMS tails are loaded and trucked from stockpiles to feed waste pass as required.

16.13.1 Crushing Facilities

The existing crusher chamber and accompanying excavations on the 1,150 mRL at Kipushi are currently being rehabilitated and will be recommissioned. The existing Crushing and Ore handling infrastructure will be replaced. KICO have ordered a Sandvik CJ615 Jaw Crusher. Sandvik have completed an analysis using inputs provided by KICO. This flowsheet can be seen in Figure 16.44.

Figure 16.44 Kipushi Ore Handling Flowsheet

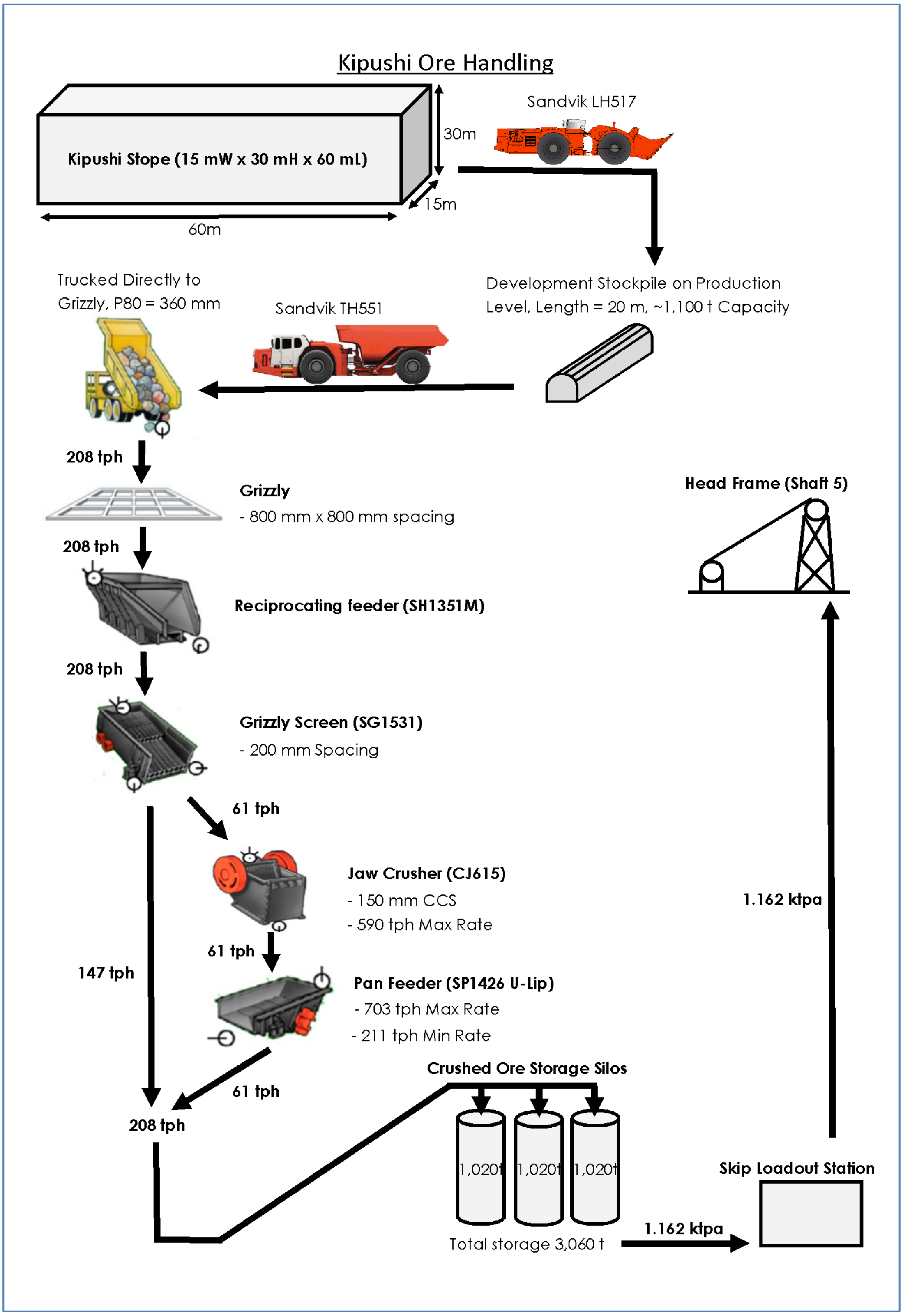


Figure by OreWin, 2017.

16.13.2 Crusher Specifications

An 800 mm x 800 mm grizzly spacing has been assumed, this has been based on similiar operations that utilise underground jaw crushers and simular rock properties, the grizzly spacing is to be confirmed in the next phase of the study. KICO requested a P_{80} for the crushing at Kipushi of 200 mm, this is within the test perimeters of the Sandvik CJ615. The Sandvik specifications for the CJ615 jaw crusher are:

- Feed Opening – 1,500 mm x 1,070 mm.
- Maximum Feed Size – 960 mm.
- Maximum Motor Power – 200 kW.
- Closed Side Setting (CSS) – (125 mm – 300 mm).
- Nominal Capacity – (385 tpa – 1,085 tpa).
- Jaw Plates - Coarse Corrugated (CC) / Sharp Teeth (ST) / Heavy Duty (HD).
- Total Weight – Approximately 53,000 kg.

Figure 16.45 1,150 mRL Crusher and Silos

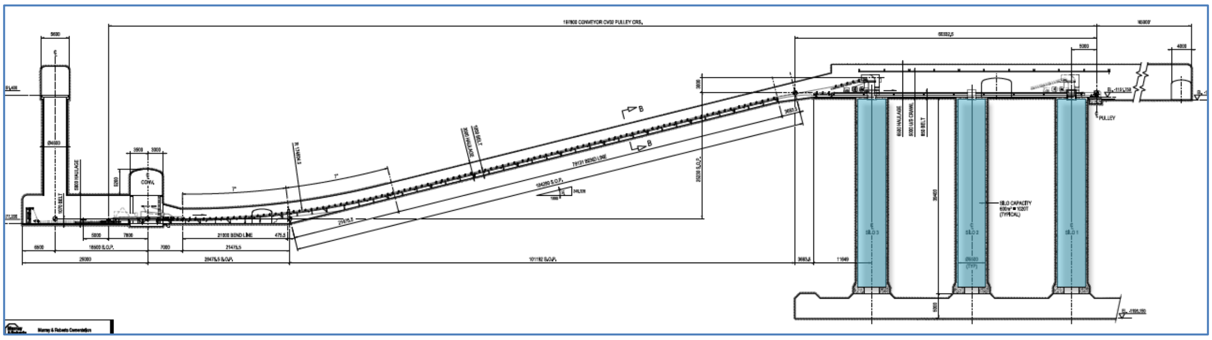


Figure by MRC, 2017.

16.14 Mine Ventilation and Cooling Design

The estimated peak airflow requirement for Kipushi is 570 m³/s. The airflow requirements are based on meeting the minimum regulatory airflow requirements for diesel exhaust dilution as set out in regulation 10.52 of the 1995 WA Mines Safety and Inspection Regulations (WAMSIRs). These regulations require a minimum diesel exhaust dilution rate of 0.05 m³/s/kW to be circulated.

With the shafts available as airways at Kipushi, the exhaust configuration options will be twin exhausts on Shafts No. 4 and No. 3. Both exhaust shaft systems should be stripped of all steelwork, including Shafts 2B, 4B, and 19, which is common to both systems. Stripping of steelwork from internal intake Shafts 1TER, 9T, and 15 is also recommended, with fan power savings of 475 kW and associated primary fan capital savings, the remaining shafts would be used as intake airways. Alternative exhaust and intake configurations were analysed but the fan duty and cooling estimate were determined based on this dual exhaust model.

Peak primary fan operating pressure is over 4,000 Pa and centrifugal fans are recommended. The modelled fan duties are shown in Table 16.45.

Table 16.45 Modelled Fan Duties

Shaft	Quantity (m^3/s)	Peak LOM Collar Total Pressure* (Pa)	Fan Shaft Power (kW)
With only Shafts 3, 4, 2B, 4B, and 19 Stripped			
No. 3	230	5100	1565
No. 4	340	4800	2175
Total	570	–	3740
With Shafts 3, 4, 2B, 4B, 19, 1TER, 9T, and 15 Stripped			
No. 3	230	4350	1335
No. 4	340	4050	1835
Total	570	–	3170

*Includes a 10% margin on modelled pressure in case mine resistance is higher than that modelled. Fan pressure will be higher due to losses in the shaft elbow and horizontal ductwork, which depend on the fan design.

The modelled cooling requirements and cooling plant design parameters are shown in Table 16.46.

Table 16.46 Modelled Cooling Requirements

Plant Design Conditions	Initial Production	Mine at Full Depth
Surface ambient	86.7 kPa, 20.4°C wb, 25.3°C db	
BAC inlet	86.7 kPa, 18.9°C wb, 25.3°C db	
Condenser cooling tower inlet	86.7 kPa, 21.9°C wb, 25.3°C db	
No. 1 Shaft		
Cooling capacity at the BAC's	1.5 MW	3.5 MW
Total intake airflow	130 m^3/s	144 m^3/s
BAC intake airflow (approx.)	36 m^3/s	84 m^3/s
Required mixed air temperature	17.1°C wb	12.8°C wb
No. 2 Shaft		
Cooling capacity at the BAC's	–	2.5 MW
Total intake airflow	–	103 m^3/s
BAC intake airflow (approx.)	–	60 m^3/s
Required mixed air temperature	–	12.8°C wb

16.14.1 Airflow Requirements

16.14.1.1 Regulatory Design Assumptions

The airflow requirements in this report are based on meeting the minimum regulatory airflow requirements for diesel exhaust dilution, as set out in regulation 10.52 of the 1995 WA Mines Safety and Inspection Regulations (WAMSIRs). The WAMSIRs are widely used across Australia as a basis for ventilation design, however, it should be noted that higher diesel exhaust dilution rates are used in other jurisdictions around the world.

16.14.1.2 Airflow Requirements

The Kipushi ventilation design uses a combination of parallel and series ventilation of activities. Primary exhaust is provided on each level. More polluting activities such as production mucking and backfill should be parallel ventilated on the level direct to exhaust. The remaining less polluting development and non-diesel activities can either be parallel ventilated, or series ventilated off the decline.

For the parallel ventilation of production mucking activities, sufficient primary airflow must be supplied to each active parallel circuit to cater for the loader and one truck on the level. The other trucks assigned to the loader are assumed to be hauling. There should be no activities scheduled downstream of the production mucking crew on a level when they are working. The airflow rate calculated in this report is designed to cater for a planned production rate of 800 ktpa.

The airflow rates applied to parallel ventilated activities are detailed in Table 16.47.

Table 16.47 Parallel Ventilated Level Airflow Allocation

Production Mucking	**kW**	**Airflow (m^3/s)**
51t dump truck	405	20.3
17t LHD	298	14.9
Total		35.2
Allowance		**40***
Backfill		**Airflow (m^3/s)**
Allowance		**30**

*Includes an additional allowance to ensure the minimum airflow requirement is met as regulator settings are never precise.

16.14.2 Primary Ventilation Circuit Design

Based on the limited airflow capacity of the vertical airways, it is calculated that the No. 4 Shaft is not large enough to handle all the exhaust airflow and a second exhaust shaft is required. The No. 3 Shaft is best suited for this purpose. The remaining Shafts No. 1, 2, and 5, remain as intake airways. In this case, a constraint is that the air velocity range of 7.0 to 12.0 m/s should be avoided to prevent water suspension issues in the exhaust shafts.

Although these shafts have been in place for a long time and would appear to be stable, splitting the ventilation mitigates some risk of water suspension, by splitting the duty between two exhaust shafts. Modelled primary fan pressures (discussed in Section 16.14.3) are higher with smaller profile shafts and centrifugal fans are required. Centrifugal fans can achieve higher pressures and typically have higher peak efficiency than axial fans. They also run slower than axial fans and are typically quieter, which would be an advantage considering the proximity of local residents around the mine. Centrifugal fans are also better suited than axial fans to wet airstreams and the primary ventilation circuit layout is shown in Figure 16.46.

Figure 16.46 Primary Airflow Circuit

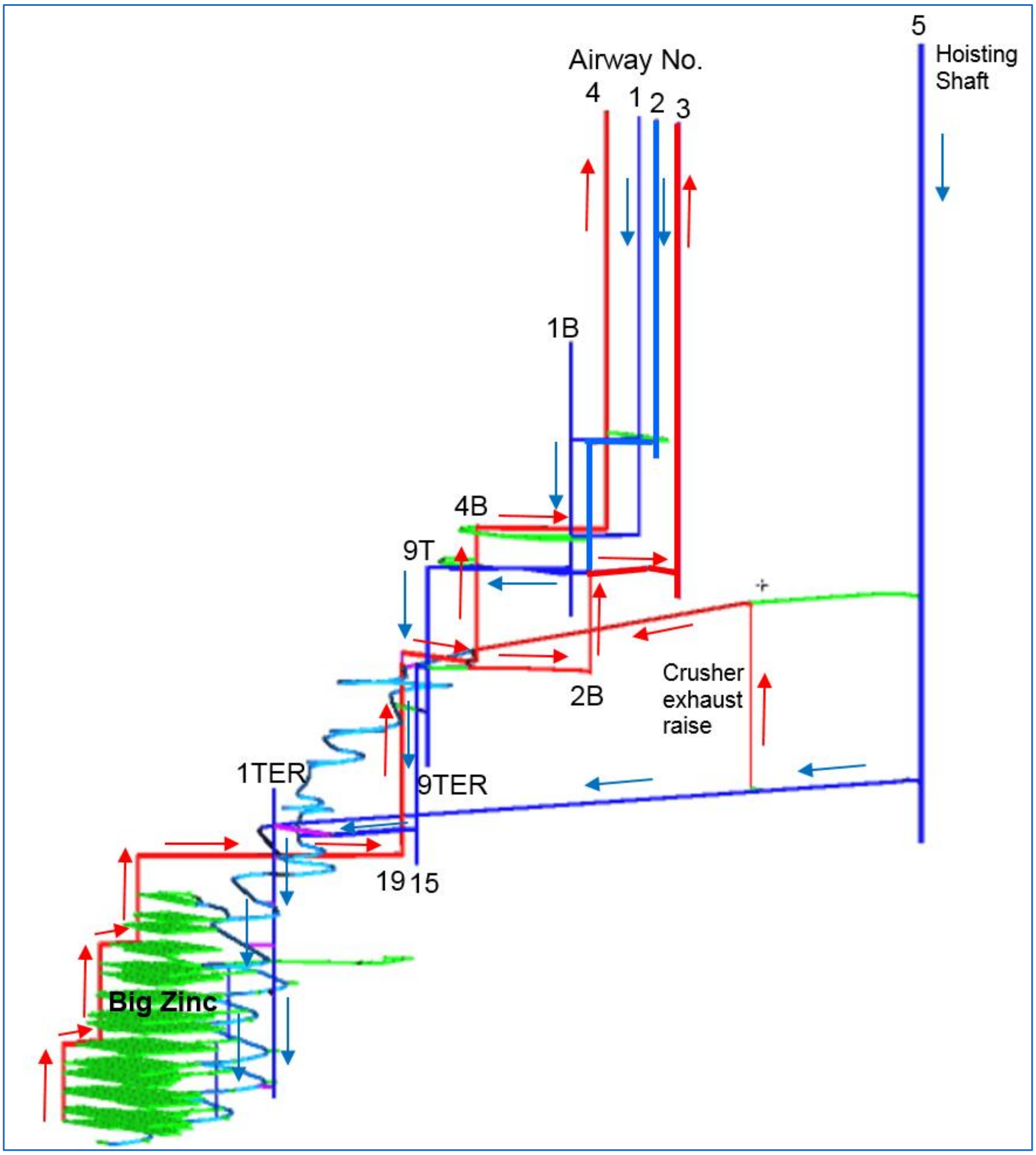


Figure by OreWin, 2017.

16.14.3 Primary Fan Duty

Design of the shaft steelwork present in all the Kipushi shafts was provided by KICO. For the fan duty modelling, it is assumed that the No. 3, No. 2B, No. 4, No. 4B, and No. 19 Shafts would be stripped of all steelwork.

The shaft steelwork information provided by KICO was insufficient to calculate the shaft resistances and so a generic friction factor of 0.025 kg/m^3 used in the modelling to estimate the resistances of the remaining unstripped shafts. More accurate calculations of shaft resistances can be made if the shaft furniture details can be supplied (beam dimensions, spacing, and profiles). Ultimately, when the mine is operating, actual shaft resistances can be measured through shaft barometric pressure surveys.

The peak Kipushi Mine airflow duty occurs from Year 2 when the fleet requirements are at a maximum. The peak pressure duty, however, occurs later in the mine life, when the mine reaches full depth. The modelled fan duty includes a 10% margin on pressure to cater for increases in the mine resistance above that modelled.

To avoid the potential for water suspension in exhaust raises, the air velocity range of 7–12 m/s is normally avoided in the shafts (although the critical velocity for this phenomenon is closer to 8 m/s). At the airflow rates required for Kipushi, airflow rates in each exhaust shaft will be marginally within the upper limits of this range. To minimise the potential for water suspension in the raises, the exhaust airflow duty was split between the two exhaust shafts roughly in proportion to the shaft cross-sectional area so that the air velocity in both shafts is roughly the same range.

16.14.4 Heat Modelling

16.14.4.1 Preliminary Cooling Plant Surface Design Temperatures

To correctly model the size of the cooling plant required for Kipushi Mine, it is necessary to determine the appropriate surface design wet bulb temperature at which the mine heat loads will be modelled. This is normally done by carefully analysing hourly site dry bulb, relative humidity, and barometric pressure data that has been collected over a number of years. The wet bulb temperature is calculated using these data and for regions like Kipushi, typically the 95th percentile wet bulb temperature would be used for the heat modelling. Unfortunately, the only data that was available from site was daily average dry bulb temperature, and daily average relative humidity data from 2014, which was not suitable for the analysis.

16.14.4.2 Heat Modelling Design Parameters

The heat modelling for this report predicts the wet bulb temperature on the Big Zinc decline. This temperature must allow for the predicted temperature rise on the levels to ensure wet bulb temperature limits are not exceeded at the workplace. For instance, activity, previous heat modelling has determined that a temperature rise of up to 4°C wet bulb can be expected at the workplace. To keep workplace temperatures below the stop job limit of 32°C wet bulb, an absolute decline wet bulb temperature limit of 28°C therefore applies (28°C + 4°C = 32°C). For the purposes of sizing cooling plants, however, decline design temperatures are normally 2°C wet bulb lower, at 26°C wet bulb. This allows some margin for the normal variations in temperature that occur underground and for changes in primary airflow rates and distribution.

The heat modelling for this report was conducted at the estimated 98th percentile surface wet bulb temperature of 20.4°C (hereafter referred to as the "surface design temperature"). The amount of cooling required to reduce the decline temperatures to between 26.0°C wet bulb (the maximum temperature for optimum workplace conditions) and 28.0°C wet bulb (the maximum temperature to avoid stop job conditions in the workplace) was then modelled.

16.14.4.3 Heat Loads

Diesel Equipment

Trucks

Heat from trucks is mainly given off while hauling. For this reason, this heat source is represented as a linear heat load in the Ventsim models. The truck linear heat load was calculated based on a production rate of 0.8 Mtpa of ore and 0.4 Mtpa of waste. It was assumed all waste is hauled and hoisted. The potential energy gained by the rock hauled is subtracted from the total energy output of the trucks and this is converted to truck linear mechanical power by multiplying by the truck efficiency (35%). Truck linear mechanical power is entered into Ventsim and the program converts this into linear heat within the program.

Loaders and Ancillary Diesel Fleet

Heat loads for the loaders and ancillary diesel fleet were determined by applying load and usage factors to calculate the average diesel mechanical power output of each machine on a continuous basis. The diesel heat loads in the form of diesel mechanical power were then applied to the model as point sources on the decline. Heat from fleet operating on the parallel ventilated levels is exhausted on the level and so does not affect the decline temperature. It was assumed in the calculations that, on average, two of the loaders are ventilated by the parallel ventilation airflow allocation and therefore was not included in the heat load calculations. Ventsim divides the mechanical power by 35% to calculate the equivalent heat load produced.

Electrical Equipment

Electrical heat loads from the jumbos and secondary fans were also applied to the models. All jumbo and secondary fan electrical power used underground was assumed to be converted to heat. As with the diesel fleet, load and usage factors were applied to calculate the average power consumption of each machine on a continuous basis. Heat from electrical equipment was applied as point heat sources on the decline. Secondary fans are responsible for most of the electrical heat load in the mine.

16.14.5 Heat Modelling Results

16.14.5.1 Initial Production

The 'initial production' heat model calculates decline wet bulb temperatures, with the first leg of the 5.5 m diameter Big Zinc RAR system developed and with 1.5 MW of cooling applied at No. 1 Shaft, are shown in Figure 16.47. With this amount of cooling applied, decline temperatures are close to the design temperature of 26°C wet bulb. This includes, in addition to the first leg of the RAR, exhaust on all four levels accessing the raise one backfill, two production and one development (bottom level).

Figure 16.47 Modelled Wet Bulb Temperatures – Initial Production 1.5 MW Cooling

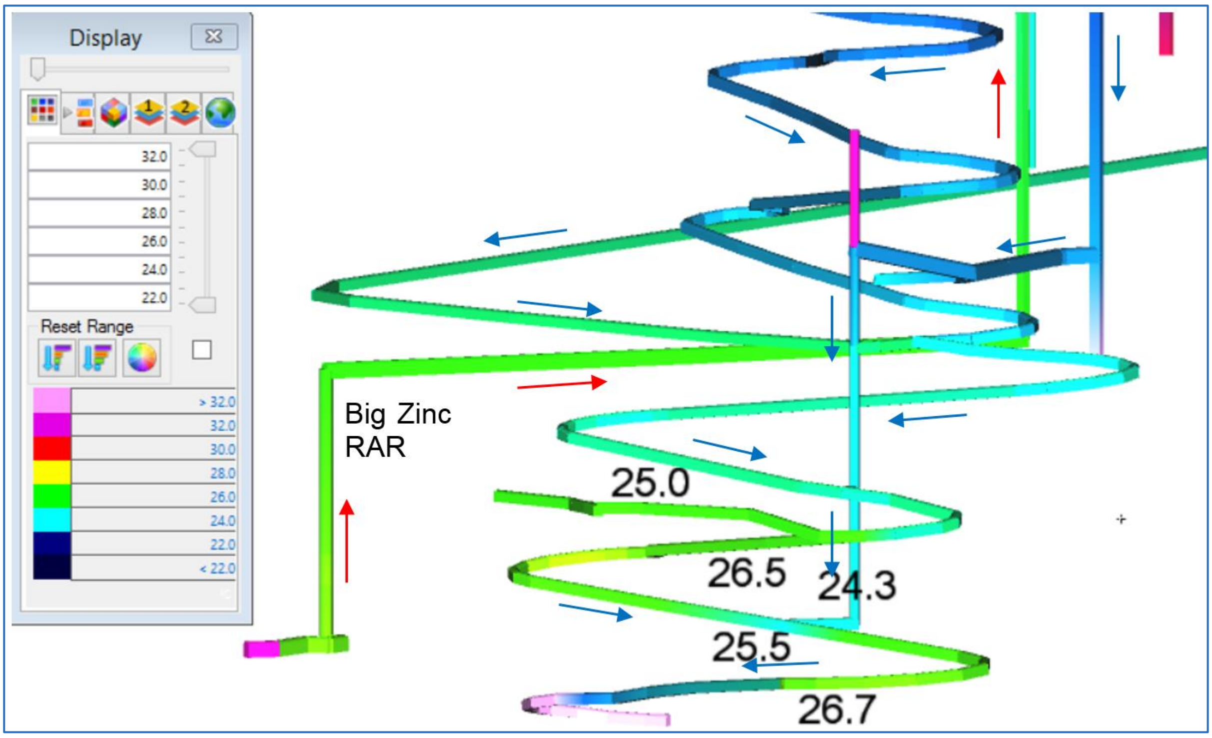


Figure by OreWin, 2017.

16.14.6 Cooling Requirements and Cooling Plant Design Parameters

To allow the cooling plant to operate efficiently over a range of temperatures, or when there are reductions in tower efficiency due to fouling, the bulk air cooler, water chiller, and condenser cooling tower designs should all incorporate a design temperature margin to ensure the cooling plant can deliver a satisfactory level of cooling, when the surface air temperature is above or below the design value of 20.4°C wet bulb. The design specifications for the cooling plant should, therefore, be based on slightly more onerous conditions than the expected operating conditions.

The design wet bulb temperature for the condenser cooling towers (incorporating the factor of safety) should be 1.5°C higher than the expected operating conditions. This will allow the cooling towers to cool the condenser water to the design temperature when surface wet bulb temperatures exceed the design 98th percentile, or when the condenser efficiency is compromised due to fouling.

Bulk air cooler (BAC) performance conversely increases with increasing surface air temperature, and the BAC design (with the design margin incorporated) should allow the full design cooling capacity to be delivered at surface wet bulb temperatures 1.5°C below the expected operating conditions.

16.15 Production Plan

The following general planning criteria were applied to determine priorities for initial production.

- Extraction of primary SLOS stopes before secondary stopes as Figure 16.29.
- Mining of SLOS extraction level before sub level.
- Mining of the Pillars only once sublevel below and extraction level above are mined.
- Highest Grade.
- Highest Productivity.
- Lowest Mining Cost.

16.15.1 Production Summary

A yearly production of 0.8 Mtpa was achieved with full production starting in Year 2021. A total of 8.851 kt of ore with an average Zinc grade of 32.14% and NSR10 value of 309$/t was scheduled to be mined during the 13-Year mine life as in Figure 16.48, Figure 16.49 and Figure 16.50. During the mine life, a total of 2,008 kt of waste will be produced.

In the ore produced from designed stopes, a significant amount of economic grade material will be produced during stope and access development. This material is included as ore in the production schedule, where the majority is defined as low-grade (NSR10≥51$/t and NSR10<135$/t) as in Figure 16.51. The planned Kipushi development and production schedules are summarised in Table 16.48 and Table 16.49.

Figure 16.48 Yearly Total Production and Average Zinc Grade

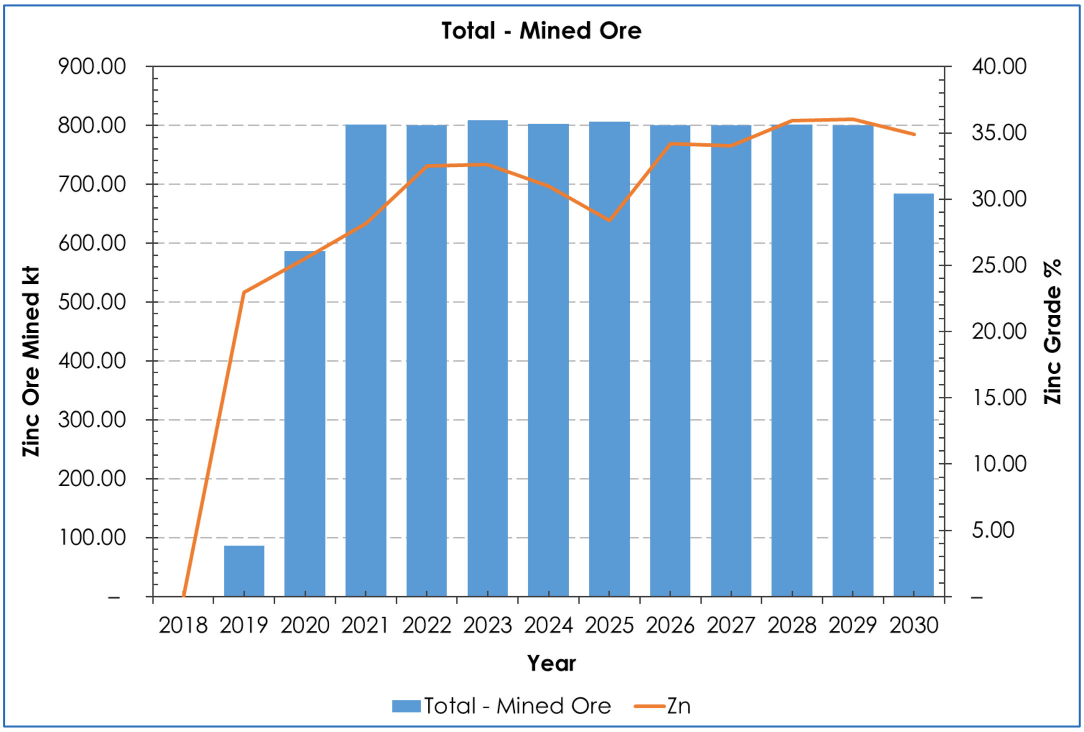


Figure by OreWin, 2017.

Figure 16.49 Yearly Total Production and Average NSR10

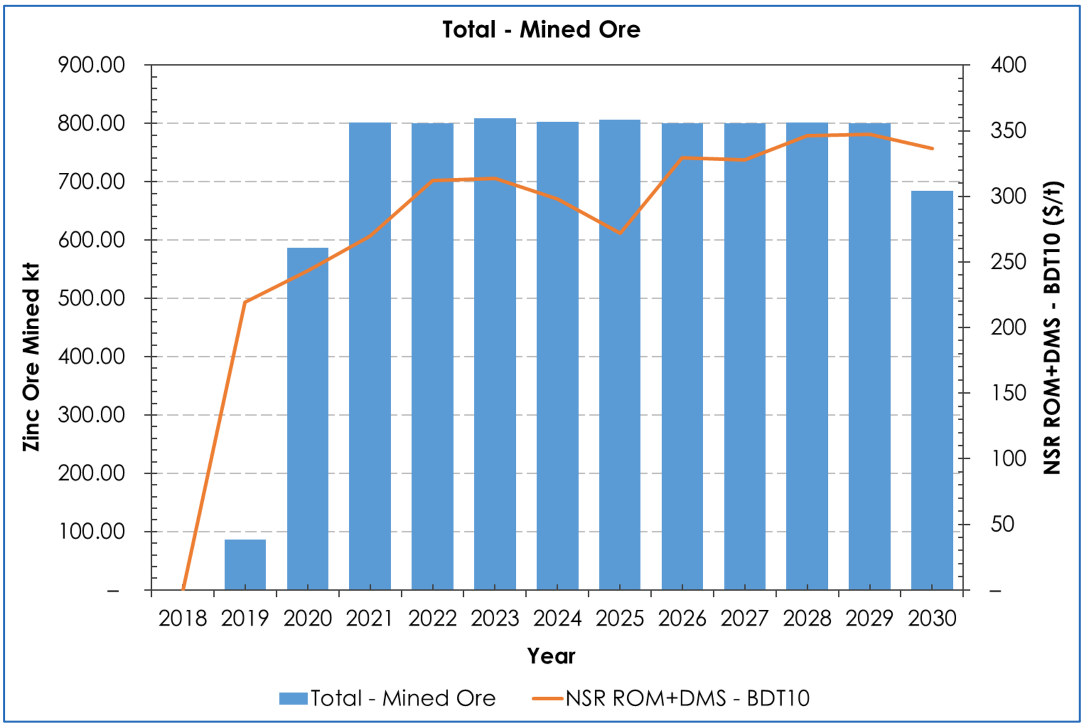


Figure by OreWin, 2017.

Figure 16.50 Total Measured and Indicated Production and Average Zinc Grade

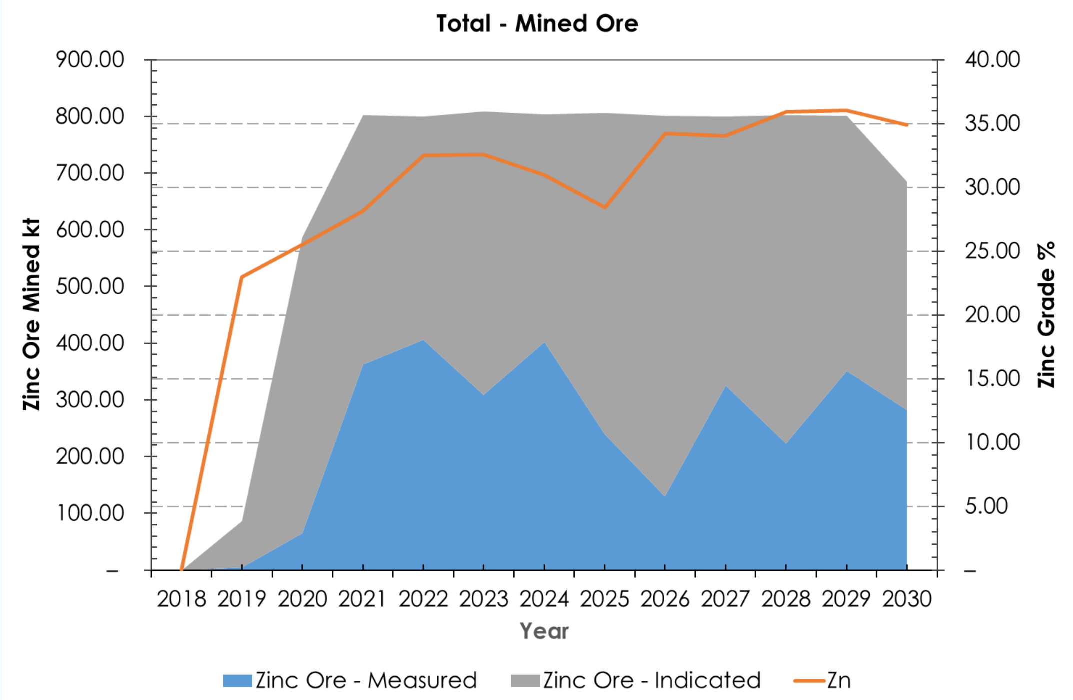


Figure by OreWin, 2017.

Figure 16.51 Total Low-Grade and High-Grade Mined Ore and Average Zinc Grade

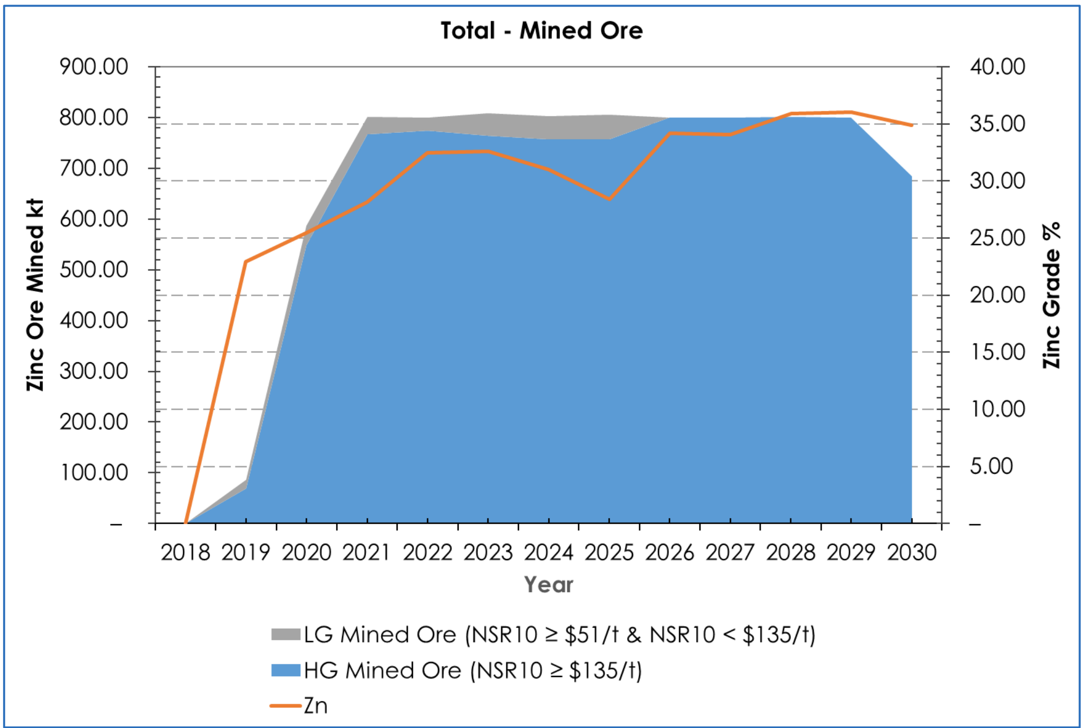


Figure by OreWin, 2017.

Table 16.48 Kipushi Development Schedule Summary

	Units	Total	Schedule Year												
			-1	1	2	3	4	5	6	7	8	9	10	11	12
Lateral Development															
Zinc Decline	m	**2,523**	167	493	177	398	418	524	345	–	–	–	–	–	–
Zinc Access	m	**6,086**	271	885	1,409	575	837	1,073	1,034	–	–	–	–	–	–
Zinc Drive 5x5	m	**2,827**	–	404	539	487	105	655	102	488	47	–	–	–	–
Zinc Drive 7x5	m	**4,068**	–	218	354	709	524	729	1,038	496	–	–	–	–	–
Zinc Drive 8x6	m	**1,309**	–	–	163	284	209	93	319	241	–	–	–	–	–
Waste Drive 5x5	m	**3,123**	–	387	443	503	139	468	248	751	185	–	–	–	–
Waste Drive 7x5	m	**4,841**	69	341	398	708	786	816	1,036	688	–	–	–	–	–
Waste Drive 8x6	m	**1,635**	–	–	107	289	280	130	345	435	49	–	–	–	–
Fresh Air Way 5x5	m	**2,557**	160	406	412	373	195	275	735	–	–	–	–	–	–
Waste Pass Access 5x5	m	**437**	90	347	–	–	–	–	–	–	–	–	–	–	–
Total Lateral Development	**m**	**29,405**	**757**	**3,481**	**4,004**	**4,326**	**3,493**	**4,763**	**5,201**	**3,098**	**281**	**–**	**–**	**–**	**–**
Vertical Development															
Ventilation Raise 4m	m	**645**	–	180	60	120	75	105	105	–	–	–	–	–	–
Pump Line Development 6"	m	**204**	–	–	61	12	58	35	37	–	–	–	–	–	–
CRF Vertical	m	**19**	–	19	–	–	–	–	–	–	–	–	–	–	–
Cement/Diesel/Waste Pilot 380 mm	m	**2,976**	1,187	1,790	–	–	–	–	–	–	–	–	–	–	–
Waste Raise 900 mm	m	**1,325**	–	626	699	–	–	–	–	–	–	–	–	–	–
Total Vertical Development	**m**	**5,170**	**1,187**	**2,615**	**821**	**132**	**133**	**140**	**142**	**–**	**–**	**–**	**–**	**–**	**–**

Table 16.49 Kipushi Production Schedule Summary

	Units	Total	Schedule Year												
			-1	1	2	3	4	5	6	7	8	9	10	11	12
Total – Mined Ore	**kt**	**8,581**	**–**	**86**	**587**	**802**	**800**	**809**	**803**	**807**	**801**	**800**	**802**	**801**	**685**
NSR ROM+DMS - BDT10	$/t	309.10	–	219.29	243.19	269.65	312.18	313.57	297.98	272.12	329.40	327.97	346.41	347.42	336.41
Zn	%	32.14	–	22.94	25.49	28.14	32.50	32.59	30.98	28.39	34.19	34.04	35.94	36.04	34.88
Cu	%	0.53	–	1.74	1.01	0.98	0.55	0.59	0.48	0.58	0.34	0.27	0.32	0.40	0.24
Pb	%	0.85	–	1.29	1.28	1.07	1.00	1.04	0.66	0.97	0.17	0.66	0.93	0.44	1.21
Sulphides	%	64.13	–	52.61	55.03	59.63	65.31	64.45	62.31	58.76	67.34	64.83	69.63	71.50	65.69
Fe	%	8.34	–	8.43	8.11	8.72	8.28	8.41	8.30	7.83	8.46	7.71	8.08	9.50	8.25
S	%	23.74	–	19.56	20.46	22.21	24.19	23.65	23.23	21.93	25.24	23.76	25.59	26.74	23.72
As	%	0.15	–	0.21	0.13	0.16	0.17	0.14	0.15	0.18	0.18	0.13	0.12	0.16	0.13
Ag	g/t	16.97	–	22.96	24.66	27.44	21.28	20.35	15.79	15.41	7.67	11.88	14.17	14.89	14.15
Ge	ppm	46.58	–	34.88	39.17	40.37	41.99	44.54	47.12	41.03	44.20	48.07	55.85	54.51	56.27
Co	ppm	13.33	–	21.58	14.81	20.32	16.14	9.35	8.69	13.12	22.19	13.67	8.98	11.59	6.27
Cd	ppm	1,586	–	1,211	1,400	1,464	1,664	1,593	1,497	1,416	1,531	1,615	1,778	1,702	1,816
Density	t/m^3	3.60	–	3.60	3.55	3.60	3.64	3.65	3.63	3.51	3.59	3.56	3.61	3.67	3.53

16.16 Hydrogeology

16.16.1 Summary

Pumping requirements were based on the 2017 Golder Hydrogeological study which shows the simulated mine inflow rates predicted for the 2016 PEA designs. The mining rate and design depth for the 2016 PEA exceeded the current rate and designs and such, the predicted inflows were used to estimate the inflow for the purposes of this study. For intermediate levels where predicted inflows weren't modelled, estimates were made based on levels that were.

At the maximum mining depth of 1,590 mRL, the maximum predicted inflow occurs at 2,808 m^3/h. The dam on the 1,112 mRL level is the closest location to the proposed Kipushi 2017 PFS designs that, in turn, dewaters to the surface. Therefore, at the maximum mining depth and inflow a dewatering pumping system is required that is capable of moving approximately 3,000 m^3/h up to 480 m of head. Figure 16.52 shows the predicted inflow by level and Figure 16.53 shows the proposed dewatering levels over the LOM.

Figure 16.52 Mining Levels and Predicted Simulated Inflow

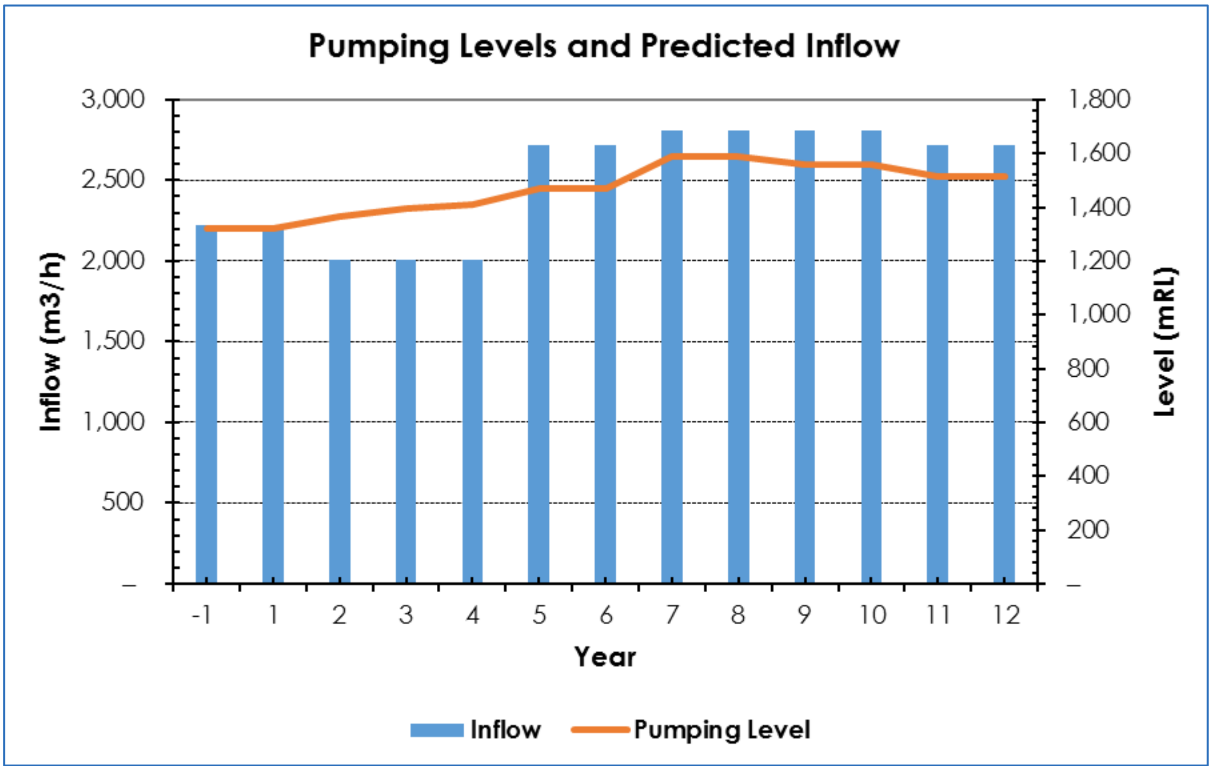


Figure by OreWin, 2017.

Figure 16.53 Proposed Dewatering Levels over LOM

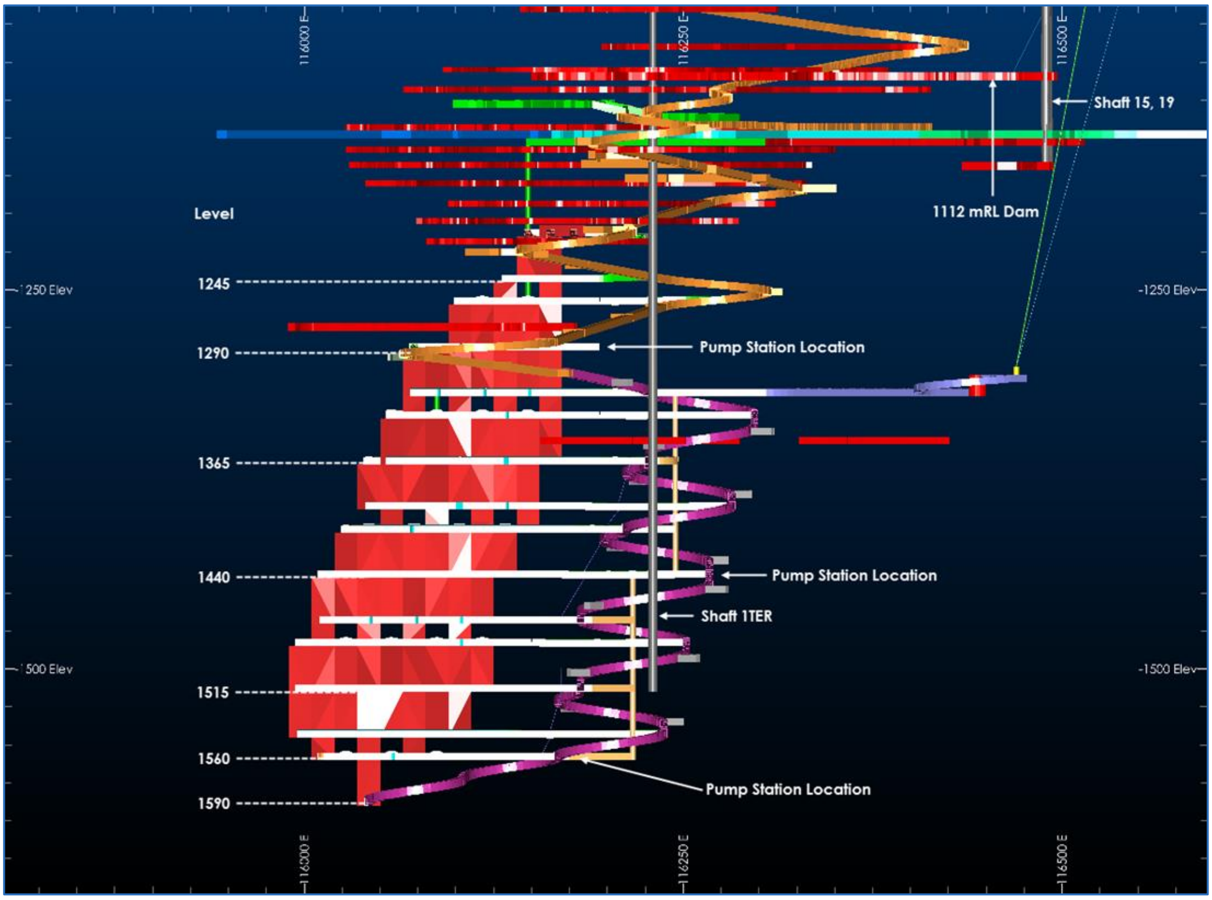


Figure by OreWin, 2017.

Figure 16.54 shows the updated water handling process flow diagram, showing the proposed workings and the pumping system to the 1,112 mRL dam. It must be noted that phase one, phase two and phase three dewatering pumping stations occur separately as mining progresses. Dewatering pumps are initially located on the 1,290 mRL level and are moved to the 1,440 mRL station as mining progresses past the 1,440 mRL level. Once mining has progressed past the 1,560 mRL level, dewatering pumps are moved from the 1,440 mRL station to the 1,560 mRL station. Additional pumps are purchased as required, as inflow increases with depth. Dewatering pumps will remain on the 1,560 mRL station for the LOM. Levels between the pumping stations will either feed the dewatering stations through the use of submersible pumps, from sumps or by gravity.

Figure 16.54 Updated Water Handling Process Flow Diagram

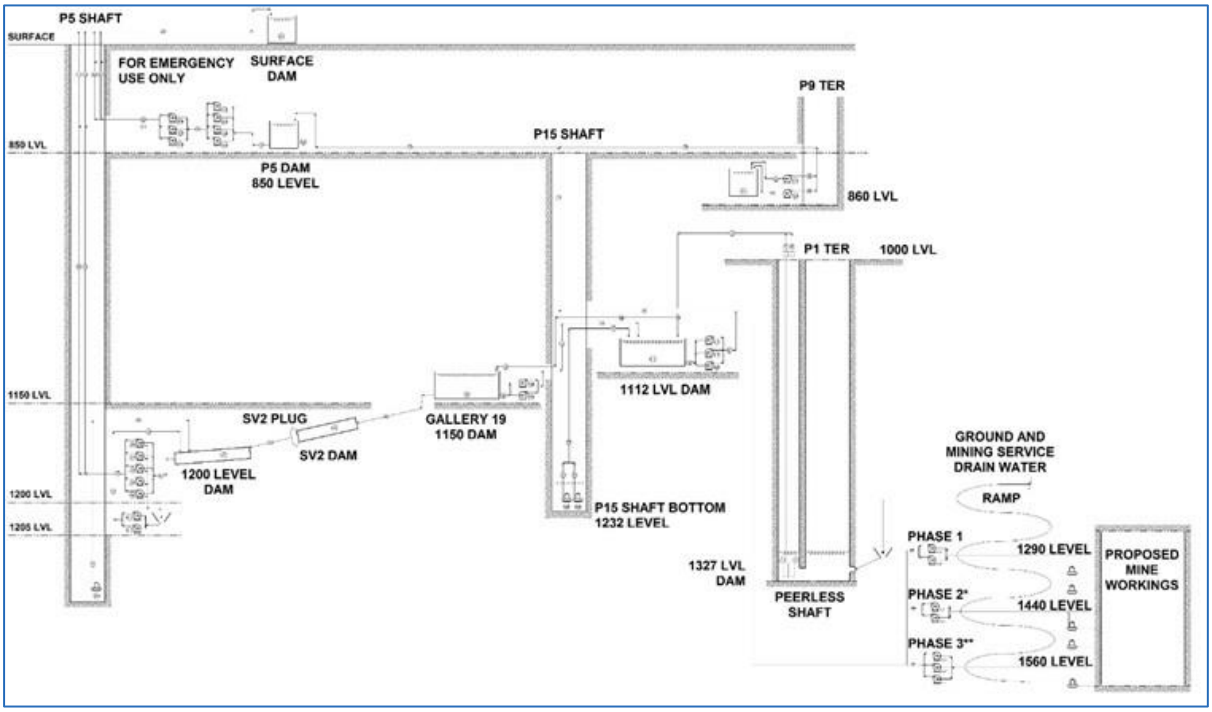


Figure by OreWin, 2017.

16.16.2 Hydrogeological Study

The inflows simulated in the base case scenario fall within the proposed pumping capability of the mine, but with no allowance for redundancy. The uncertainty in the inflow volumes calculated could be reduced through undertaking relevant aquifer tests of the shallow and deep aquifers.

Table 16.50 Simulated Mine Inflow Rates 2017–2027

Year	Maximum Depth of Mining (mamsl)	Predicted Inflow Rate (l/s)
2017	200	618
2018	-90	491
2019	-90	557
2020	-165	755
2021	-240	780
2022	-315	799
2023	-315	812
2024	-285	807
2025	-495	831
2026	-480	826
2027	-180	822

Predicted maximum inflow of the Kipushi 2017 PFS design, as shown in Section 16.16.1 is 2,808 m^3/h, on the 1,590 mRL level. During the LOM, multiple active mining levels are operational, with up to 4 stopes or pillars being extracted simultaneously. Submersible pumps located on the active mining levels will pump to dewatering stations positioned off the decline. Dewatering pump stations will initially be located on the 1,290 mRL, then as mining progresses on the 1,440 mRL and lastly the 1,560 mRL.

Based on the pump specifications, two centrifugal dewatering pumps would be required on the 1,290 mRL to dewater to the 1,112 mRL dam. When mining reached the 1,440 mRL the pumping station would be moved to this level and again, two pumps would be required to feed the 1,112 mRL dam. Finally, when mining reaches its full depth at the 1,590 mRL, the pumping station would be moved to the 1,560 mRL, where three pumps would be required to meet quantity and head requirements to feed the 1,112 mRL dam. Table 16.51 shows the specifications of a centrifugal dewatering pump which would meet the demand.

With stopes and pillars being mined simultaneously on multiple levels at a time, water inflow from exposed faces must be managed via the use of submersible pumps, located in sumps feeding the decline dewatering stations. Gravity feed will also be employed where convenient, depending on the level being mined. Based on scheduling and varying active mining levels, it was calculated that 8 pumps would be required for LOM.

Table 16.51 Pump Specifications

	Units	Value
Power Consumption	kW	2,000
Power Frequency	Hz	50
Capacity	m^3/h	1,000
Efficiency	%	82.8
RPM	rpm	1,490
Head	m	480

17 RECOVERY METHODS

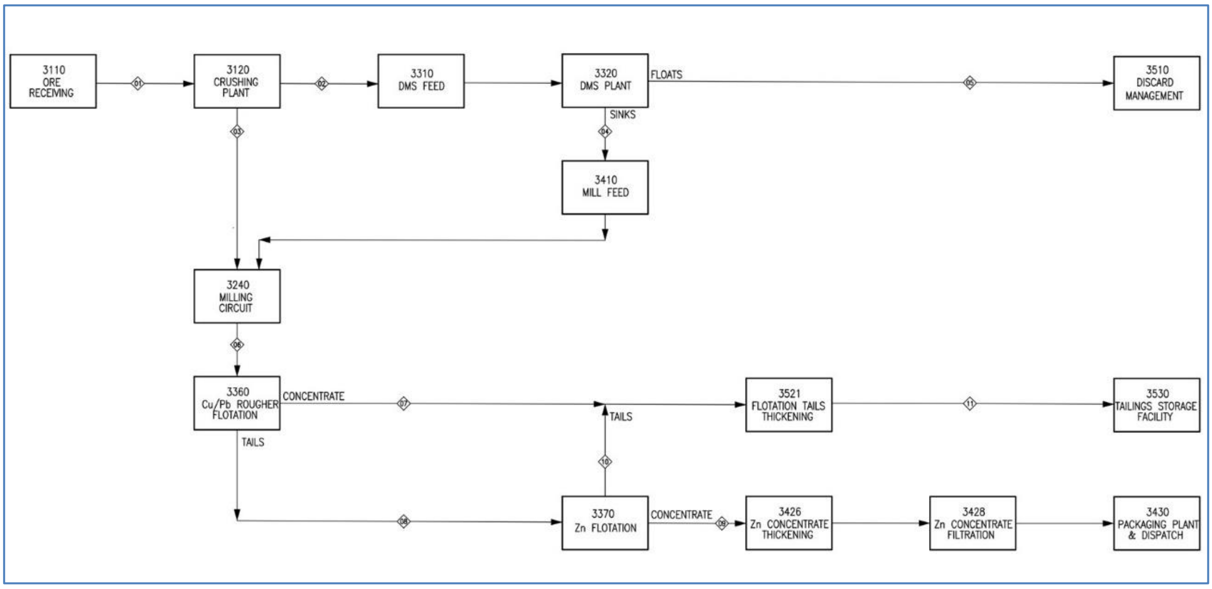
This section has not been changed from the Kipushi 2017 Prefeasibility Study and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Kipushi 2017 PFS.

17.1 Overview

The Kipushi 2017 PFS process plant has a name plate capacity of 800 ktpa and the life-of-mine average annual planned zinc concentrate production is anticipated to be 381 ktpa, with a concentrate grade of 59% Zn. Total zinc production is anticipated to be 8.6 Mt ore at 32.14% Zn over a period of eleven years to produce 2,472 kt of zinc metal in concentrate.

The proposed process plant consists of two stage surface crushing and screening to produce a top size of 20 mm. The screened -20+1 mm material will be subjected to Dense Media Separation at a density cut point of 3.1 t/m³ which will reject the low density dolomitic material as tails. The heavy base metal sulphides will concentrate in the sinks and be combined with the screened -1 mm as feed to the milling circuit. The milling circuit consists of a single ball mill in closed circuit with a cyclone cluster and will grind the material to 80% passing 106 µm. The milled slurry will be conditioned with reagents for copper and lead rougher flotation, and the tails will again be conditioned with reagents suitable for zinc flotation. Zinc flotation concentrate will be thickened, filtered and bagged for loading onto train wagons, ready for dispatch to the market. The Cu/Pb concentrate is combined with zinc float tails, thickened and pumped to a new tailings storage facility. The DMS discard is stockpiled and used for cemented rock fill. The proposed circuit block flow diagram is illustrated in Figure 17.1.

Figure 17.1 Process Plant Block Flow Diagram



17.2 Process Plant Design

17.2.1 Design Basis

Process plant design as presented herein, was based on:

- The mine production schedule.
- Testwork results conducted by Mintek Laboratories.
- The associated Process Design Basis file issued by KICO.
- Utilising proven and established process technologies; with a bias towards modular plant systems.
- Laying out the plant within the constraints of a brown field site.

Where testwork information was not available, assumptions were made. A statistical analysis of the production schedule feed grades is presented in Table 17.1.

Table 17.1 Plant Feed Grades

		Grades		
Element	**Units**	**Min.**	**Avg.**	**Max.**
Zn	%	22.94	32.05	36.04
Cu	%	0.24	0.54	1.74
Pb	%	0.17	0.85	1.29
Fe	%	7.71	8.34	9.50
S	%	19.56	23.70	26.74

The plant design is based on the LOM weighted average grades. As the project moves forward to the FS, grade variability will be defined at a more granular level in the mine plan and, will be utilised in combination with the FS variability testwork to refine the design of the plant.

17.2.2 Design Assumptions

Testwork data has been used for the design principles, however some design assumptions have been made as follows:

- A crusher work index of 15.1 kwh/t was used in the crusher simulations.
- Limited to no clays are present in the ROM feed.
- For the purpose of sizing the ball mill, it was assumed that the DMS concentrate would exhibit similar milling characteristics to the ROM ore. Ore and concentrate solids specific gravities, were derived from a mineralogical interpretation of the streams using METSIM®.
- A fesi consumption of 0.3 kg/t was assumed.
- Ball consumption in the mill was assumed to be, 1 kg/t of fresh mill feed.
- In the design of the thickeners (concentrate and tailings), the following parameters were assumed.
- Solids flux rate (0.6 $t/m^2/h$); rise rate (4 m/h); and 50% U/F solids density.
- Flocculant consumption rates are in line with industry norms.
- In the design of the filters, a flux rate of 270 $kg.m^{-2}.h^{-1}$ and a final filter moisture content of 12% was assumed.
- The use of underground water in an operating plant does not impact process performance.
- The concentrate produced is free flowing, non-reactive and does not age harden.
- Mass flow is achieved in all bins.
- Concentrate packaged into 1.8 t bulk bags is acceptable to the customer, and a single commercial sample is required per 44 t batch.

17.2.3 Design Parameters

Key parameters used in the development of the plant design and operating costs are summarised in Table 17.2.

Table 17.2 Key Design Basis Parameters

Design Basis	Units	Value
Annual throughput	ktpa	800
Crusher/DMS plant availability	%	75
Milling/float plant availability	%	91
Power availability	%	95
Overall plant availability	%	86.5
Plant Design Head Grades (LOM Weighted Average)		
Zn	%	32.14
Fe	%	8.34
S	%	23.74
Plant Feed Split		
Crusher product (-20 mm +1 mm)	% of ROM	87.2
Crusher fines (-1 mm)	% of ROM	12.8
Critical Sizes		
RoM top size	mm	200
DMS plant feed	mm	-20 to + 1
Mill feed F_{80}	mm	16
Mill product P_{80}	µm	106
Process Plant Design Parameters		
HLS testwork zinc recovery	%	98.0 to 99.7
DMS plant zinc recovery (interpolated)	%	97
DMS cut-density	t/m^3	3.1
Testwork residence time scale-up factor	number	2.5
Cu/Pb float pH		9.5
Cu/Pb concentrate mass pull	%	16
Zn float pH		11.5
Concentrate mass pull	%	67.3
Zinc concentrate grade	%	58.9
Filter cake moisture	%	12
Reagent Consumption- Based on ROM Feed		
Flocculant	g/t	30
Sodium carbonate	g/t	800
Zinc sulphate	g/t	800

Design Basis	Units	Value
Sodium cyanide	g/t	400
Sodium ethyl xanthate	g/t	20
MIBC	g/t	50
Hydrated lime	g/t	1800
Copper sulphate	g/t	1800
Sodium isopropyl xanthate	g/t	120

17.3 Process Description

17.3.1 Ore Receiving

Ore and waste is crushed underground to a P_{100} of 200 mm and hoisted to surface using the refurbished Shaft 5.

Both crushed ore and development waste will be intermittently (and separately) hoisted to surface, depositing into a single bin on surface, within the Shaft 5 headframe. Material is reclaimed from the bin via a vibrating feeder, which ultimately deposits onto a single 900 m overland conveyor connecting Shaft 5, to the main mine area at the Old Kipushi Concentrator (OKC).

The overland conveyor via a three-way transfer system either feeds a:

- ROM ore strategic/operational stockpile.
- An intermediate transfer stockpile for development waste.
- Or, the crusher plant feed bin.

17.3.2 Crushing Plant

Ore is fed to a two-stage crushing plant at a rate of 122 tph. The crusher circuit design has been set up to minimise the production of fines (-1 mm). To this end, an open circuit secondary crusher (132 kW) is used in conjunction with a closed-circuit tertiary crusher (132 kW).

Crushed material is combined on a classification screen. Oversize (+20 mm) is transferred to the tertiary crusher, whilst middlings (-20 mm to +1 mm) are transferred to the DMS plant. Screen fines (-1 mm), representing 12.8% of the ROM, feed is combined with water and pumped to the mill sump.

Coarse area spillage will be collected manually and transferred to the crushed ore conveyor feeding the classification screen, whilst fine spillage/slurry is pumped to the classification screen for recovery.

Dust suppression points are available for dust control around the secondary crusher screen and crusher operations.

17.3.3 DMS Plant

Crusher classification screen middlings are transferred via conveyor to the DMS plant feed area, from where material is fed to the DMS plant at a constant controlled rate.

The DMS plant uses atomised ferrosilicon as "medium", with a plant cut-point density of 3.1 t/m^3. The DMS feed grade is 32.14% Zn and is upgraded to ~47% Zn in concentrate at an average mass pull of ~70% to product.

DMS cyclones are used to concentrate the zinc ore, with concentrate reporting to the cyclone underflow (referred to as sinks), and the lighter minerals passing through the cyclone overflow/underflow (referred to as floats). Each cyclone stream is subsequently screened to ensure the media (FeSi) is washed and recovered from the ore streams. The washed DMS product is transferred via a conveyor to the mill feed bin. The DMS residue from the floats screen is transferred to the waste handling area.

The DMS media density is controlled with densifiers and magsep drums. Media make-up for lost or used-up FeSi is done manually.

DMS effluent is pumped to the flotation tailings thickener for process water recovery.

17.3.4 Milling Circuit

DMS concentrate from the sinks screen oversize is transferred to the mill feed bin. The mill is fed at a controlled rate, with steel balls added manually onto the mill feed conveyor. Crusher fines from the classification screen are pumped into the mill discharge sump.

The DMS concentrate and crusher fines are milled in a closed-circuit variable speed ball mill, with cyclone classification. The milling circuit comprises a single 900 kW ball mill, which has an inside diameter of 3.4 m and a length of 5.3 m. The milling circuit is designed to achieve a P_{80} of 106 µm. The cyclone overflow gravitates to the flotation circuit at a solids density of 30%.

17.3.5 Flotation

To reduce iron, lead and copper levels in the final zinc concentrate produced to acceptable levels, copper/lead and iron are removed sequentially in two stages of flotation.

In stage one, mill product (P_{80} = 106 µm, 43% Zn), feeds the copper/lead flotation circuit, where copper/lead are preferentially floated at a pH of 9.5 in four 10 m^3 tank cells in series.

The copper/lead circuit tails are conditioned prior to being pumped to the zinc flotation circuit, where zinc is preferentially floated and pyrite depressed at a pH of 11.5, in five 20 m^3 tank cells in series.

Flotation concentrate from the copper/lead circuit, is combined with tails from the zinc circuit, and pumped to the TSF, whilst the zinc concentrate produced is thickened, and then filtered, before being transferred to the bulk bag packaging facility.

The copper/lead flotation circuit uses zinc sulphate and sodium cyanide, to suppress the zinc sulphide and sodium ethyl xanthate to collect copper/lead minerals.

The copper/lead flotation tails (zinc sulphide bearing stream) is activated with copper sulphate and sodium isopropyl xanthate is used to collect the zinc sulphide.

MIBC frother is used in both circuits to assist with stable froth formation, while sodium carbonate and lime are used for pH adjustment.

17.3.6 Concentrate Handling

From zinc flotation, the zinc concentrate is pumped to the concentrate thickener, with thickener underflow (50% solids) pumped to the filter feed tank.

From the filter feed tank, slurry is pumped to either one of two fully automated vertical tower filter presses, to produce a saleable filter cake containing not more than 12% moisture.

Thickener overflow and filter filtrate are recovered to the process water circuit, whilst the filter cake is conveyed to the concentrate packaging facility at the rail siding, via a 250 m long transfer conveyor.

The configuration of the packaging facility is governed by the assumption that a single sample is required per 44 t of concentrate produced. Rather than sampling 24, 1.8 t bulk bags per batch and producing a combined composite sample, it was decided to take a falling belt sample as the tripper conveyor discharge into the respective bins. A discrete 10 to 20 kg sample is obtained per batch/bin (44 t).

As currently configured, two silos will be on line at any one time filling bags, whilst the other two will be on a sequential filling cycle. Bins are filled on a discrete batch basis to facilitate the sampling requirements.

17.3.7 Waste Handling

Waste handling includes development waste from underground mining operations, DMS residue as well as flotation tails which are thickened and pumped to the tailings storage facility. Thickener underflow is sampled for metallurgical accounting purposes, at the point where it discharges into the final tailings transfer tank.

Coarse waste (mine development waste and DMS residue) will be used for mine backfilling. The material will be stockpiled as received and transferred from the temporary stockpiles to long term stockpiles, utilising mobile equipment by a third-party contractor.

17.3.8 Utilities

17.3.8.1 Water

Raw water from the dewatering operations is received in the plant via the water reservoir tower at Shaft 5. Water not used in the plant is diverted and discharged via the existing drainage system.

From the Shaft 5 water tower, gravity based, raw water take-off is available for mobile equipment (water bowzers, fire wagons) and for the plant's raw water supply tank (500 m^3), which will also double as the firewater tank.

From the raw water tank, water is pumped to those reagent areas requiring raw water, as well as filtered gland service water for slurry pumps. Raw water is also used as a process water make-up when required.

Potable water is received from the local municipal supply and stored in the plant in a potable water tank (50 m^3) for further water distribution.

The process water tank (74 m^3) receives water from: thickener overflows, and the raw water make-up system. Process water is filtered for spray water and pumped to screens and flotation cells and where applicable used for reagent make-up.

17.3.8.2 Air Services

A duty and standby compressor system for the plant instruments, as well as for the concentrate filter presses, has been allowed for in the Kipushi 2017 PFS plant design. Flotation air blowers are used to supply air to the forced air flotation cells.

17.3.8.3 Reagents

Sodium Carbonate

Sodium carbonate (99.8%) is delivered to site in 1 t bulk bags. The bags are transferred from storage to the mixing area by forklift. Sodium carbonate is mixed with water. Once a mixed batch is finished, the solution is transferred to the dosing tank and pumped to the ball mill for dosing. Approximately, 1.5 t of sodium carbonate is used per day.

Zinc Sulphate

Zinc sulphate is delivered to site in 1 t bulk bags. The bags are transferred from storage to the mixing area by forklift. In the mixing area, zinc sulphate is mixed with water in batches. Each batch is transferred to the dosing tank for distribution. Approximately, 1.5 t of zinc sulphate is used per day.

Sodium Cyanide

Sodium cyanide (NaCN) is delivered to site in 900 kg bulk bags. The cyanide is mixed with process water to make up a 20% strength solution. On completion of dissolution, the cyanide solution will be pumped to the cyanide storage tank for distribution. Approximately, 0.8 t of NaCN is used per day.

Sodium Ethyl Xanthate

Sodium Ethyl Xanthate (SEX) is used as a sulphide flotation collector, targeting copper and lead. It will be supplied in in 850 kg bulk wooden crates. The xanthate is mixed with water to achieve a dilution of 20%. Xanthate is then pumped to the dosing tank and distributed for dosing as required. Approximately, 40 kg of SEX is used per day.

Frother (MIBC)

Frother is received in 1 m^3 ISO containers. A container is off loaded at a designated area close to the flotation plant. The ISO container will be hooked up to a dosing pump to feed frother to the flotation circuit. Each flotation circuit will have its own container and dosing pump. Approximately, 90 kg of MIBX is used per day.

Lime

Hydrated lime ($Ca(OH)_2$) is delivered to site in 1 t bulk bags. The bulk bags are moved to the make-up area by forklift. The lime powder is discharged into the lime make-up batch silo. From the lime silo, lime is metered into the agitated mixing tank and mixed with water to 20% dilution. Once a batch is made, the lime solution is transferred to the lime dosing tank for distribution. Approximately, 3.9 t of lime is used per day.

Copper Sulphate

Copper sulphate is delivered to site in 1 t bulk bags. The bags are transferred from storage to the mixing area by forklift. Copper sulphate is dissolved in water. Once a mixed batch is finished, the solution is transferred to the dosing tank for further distribution. Approximately, 3.4 t/d of copper sulphate is used.

Sodium Isopropyl Xanthate

Sodium Isopropyl Xanthate (SIPX) is used as a sulphide flotation collector, targeting zinc sulphide (sphalerite). It will be supplied in 850 kg bulk wooden crates. These will be transported from the storage yard to the SIPX offloading area and discharged to the SIPX mixing tank manually and mixed with water. Diluted SIPX is transferred to the storage tank for further distribution. Approximately, 170 kg of SIPX is used per day.

Flocculant

Bags of flocculant (25 kg) are transported from the store to the make-up area by forklift. Flocculant is manually dosed into a hopper. From the hopper, the flocculant powder is drawn by screw feeder and fed to an eductor where water is added. Flocculant is mixed to a flocculant strength of 0.5%. After the required hydrolysis time, the activated flocculant is pumped to the flocculant storage tank. Flocculant is pumped at a controlled flow rate directly to the thickener feed boxes with final dilution in the thickener areas. Approximately, 41 kg of flocculant is used per day.

17.4 Plant Ramp-up

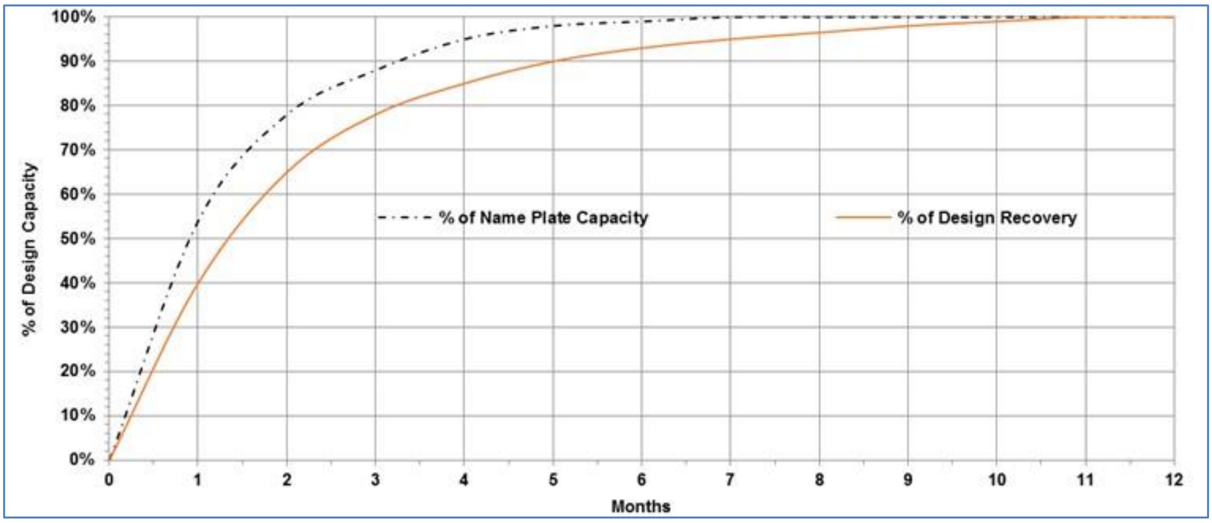
Mining activities will begin before the process plant is constructed or ready to receive ore and thus, strategic ore and waste development stockpiles have been allowed for in the design of the surface infrastructure. The plant's ramp up / strategic stockpile size is constrained by available space and is thus limited in size, to one month of storage capacity at the plant's design throughput.

- A plant-ramp up profile for ROM throughput and the concentrate produced, has been developed for the Kipushi 2017 PFS, using a typical McNulty type ramp-up curve for a relatively simple plant. The proposed ramp-up curve is illustrated (Figure 17.2).

Once the plant is constructed and cold commissioning is completed, it is estimated that:

- After two months, the plant should be able to meet 80% of its ROM nameplate capacity consistently. Full throughput should be consistently achieved after five months of full operation. That is, if there is feed.
- Design zinc recoveries should be achieved consistently in month eleven. Zinc recovery may be impacted by:
  - Water quality and reagent consumption and control strategies.
  - The low zinc grade in the first year of operation.
  - Zinc grade variability.
- The plant design is based on a utilisation of 86.7%, and thus there is some capacity for catch-up built into the design. Whilst a normal utilisation of 91% to 92% could be expected for a mill/float circuit, it is relevant to note that there is limited buffering capacity between plant sections and the power supply reliability is lower than average.
- The ramp-up strategy, taking cognisance of possible constraints and buffering capacity requirements along the entire mining value chain, will need to be revisited during the FS.

Figure 17.2 ROM and Product Ramp-Up from Start of Hot Commissioning



17.5 Mass Balances and Production Schedule

For the Kipushi 2017 PFS design basis the relevant data has been summarised and is presented in Table 17.3, whilst the corresponding mining and concentrate production schedule by year is presented in Table 17.5.

Table 17.3 Nominal Mass Balance

Description	Units	Value	Comment
Ore mined	kt	8,581	Life-of-Mine
DMS discarded	kt	2,174	Life-of-Mine
Tailings deposited	kt	2,112	Life-of-Mine
Concentrate produced	kt	4,226	Life-of-Mine
Zinc metal produced	kt	2,489	Life-of-Mine
Life-of-mine	months	130	or 11 years
Plant throughput	ktpa	800	
Zinc head grade	%	32.14	Mine Plan - Average life-of-mine grade
Plant throughput	tph	105.29	Normalised on milling plant availability
Zinc recovery	%	90.24	Assumed to be constant over LOM (Steady state)
Concentrate production	tph	51.82	
Concentrate moisture	%	12.00	
Concentrate zinc content	%	58.91	

Consideration has been given to the possible impact of grade variability on the design and operation of the plant. It is important to note that this cannot be formalised until the mine plan is developed and reported at a more granular level and the variability testwork that forms part of the FS is undertaken.

Notwithstanding this, an attempt has been made using METSIM to define the impact of grade variability on plant design and operation.

Three different scenarios have been modelled, namely: low zinc (22.9% Zn); average zinc (32.14% Zn) and high zinc (37.1% Zn). It is important to note that the zinc grades are annual weighted average values, and grades seen by the plant within any given year, may be higher or lower than that indicated.

The results of the METSIM simulation are summarised in Table 17.4 below.

Table 17.4 Possible Range of Plant Operational Scenarios

Description	Units	Value (annual data)
Shaft hoisting rate (- 200 mm @ 5% H2O)	ktpa	1,800
RoM production rate	ktpa	800
Crushing / DMS - availability	%	75 / 75
Crushing / DMS - feed rate (nom.)	t/h	122 / 106
Milling - availability	%	86.7
Milling - feed rate (nom.)	tph	75
RoM Zn grade (min./nom./max.)	%	22.94 / 32.14 / 36.04
RoM Cu+Pb grade (min./nom./max.)	%	0.41 / 1.37 / 3.03
RoM Fe grade (min./nom./max.)	%	7.71 / 8.34 / 9.5
Dolomite content (min./nom./max.) - interpolated	%	28.50 / 35.87 / 47.39
Process zinc recovery (nom.)	%	90.2
Concentrate zinc grade (min./nom./max)	%	57.44 / 58.91/ 59.54
Concentrate min./nom./max.	ktpa	236 / 384 / 437
Concentrate min./nom./max. @ 12% H2O	ktpa	268 / 436 / 497
Bags (1.8 t/bag) min./nom./max.	bags/d	603 / 786 / 895
Bags (1.8 t/bag) min./nom./max.	bags/h	25 / 33 / 37

Table 17.5 Processing Schedule

		Year Date	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030
Description	**Units**	**Total/ Year**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**
Plant Feed	kt	8581	673	802	800	809	803	807	801	800	802	801	685
Zn Grade	% Zn	32.14	25.17	28.14	32.50	32.59	30.98	28.39	34.19	34.04	35.94	36.04	34.88
Zn Concentrator Recovery	% Zn	89.61	80.00	90.24	90.24	90.24	90.24	90.24	90.24	90.24	90.24	90.24	90.24
Zn Concentrate Grade	% Zn	58.91	57.44	58.09	58.94	58.95	58.66	58.14	59.23	59.21	59.52	59.54	59.35
Zn Concentrate Produced	kt (dry)	4,196	236	350	398	403	383	355	417	415	437	437	363
Zn Produced	kt	2472	135	204	235	238	225	207	247	246	260	260	216
Zn Produced	Mlb	5449	299	449	517	524	495	456	545	542	573	574	475

17.6 Comments on Section 17

Testwork completed in 2018 will be used to update the flowsheet, optimise zinc recovery and improve understanding of the interrelation between DMS, Milling and Flotation. Issues being explored include:

- Reducing the DMS Feed size to 12 mm from 20 mm so that the DMS concentrate topsize is more appropriate as ball mill feed. The reduced topsize increases the fines proportion of the crushed ore and this increases the mass fraction reporting to flotation feed. However, milling efficiency and grind consistency is improved significantly.
- Smplifying crushing and screening
- Modifying the handling of fines and flotation feed in the circuit to optimise circuit flexibility
- Modifying stockpiles and plant layout to improve operability
- Removal of the Pb/Cu flotation stage as this results in excessive zinc losses. The previously lost zinc, together with relatively low levels of Cu and Pb, report to final zinc concentrate for what is indicated to result in a net revenue increase. An added benefit is that capital cost and reagents associated with the Pb/Cu float (such as cyanide and the Pb/Cu collector) are no longer required.

- In the opinion of the Process QP, none of the emerging information from the FS metallurgical test program nor the contemplated design changes have a material negative effect on the economics of the project. Given what are historically high feed grades for a zinc project of this magnitude, it is also considered that economic improvements arising from these contemplated changes may be under the threshold to be material. The main benefit of the changes is risk reduction, thereby improving the technical robustness of the project..

18 PROJECT INFRASTUCTURE

This section has not been changed from the Kipushi 2017 Prefeasibility Study and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Kipushi 2017 PFS.

18.1 Project Infrastructure Summary

The Kipushi Project is located within the town of Kipushi in the south-western part of the Haut-Katanga Province in the DRC and adjacent to the border with Zambia and shown in Figure 18.1. The Kipushi town is situated approximately 30 km south-west of Lubumbashi, the capital of Haut-Katanga Province. Kipushi is connected to Lubumbashi by a paved road. The closest public airport to the Kipushi Project is at Lubumbashi where there are daily domestic, regional and international scheduled flights. As part of the Project, the 34 km rail spur connecting the Kipushi Station to Munama will be reinstated to facilitate transport of concentrate.

Shaft 5 and the surface infrastructure associated at the OKC (location of the proposed processing plant) reside within two separate and discrete, fenced areas in the town of Kipushi. The two demarcated areas are linked by an existing pipe rack and an underground cable tunnel that crosses through 300 m of public space, and over one public road.

The project infrastructure relates to the surface component of operational support systems, covering all mine equipment and associated buildings, outside of what has already been defined as part of mining and processing directly responsible items.

The large site, two distinct different working areas, its historic brownfield nature and its tight enclosure within the town of Kipushi, make infrastructure more complicated than many other typical mining operations.

The property hosts surface mining and processing infrastructure, a mineral processing/beneficiation plant, offices, workshops, stores, and connection to the national power grid. All of the surface infrastructure is owned by Gécamines and is either ceded or leased to KICO. Key aspects of the project infrastructure are:

- Electricity is supplied by the state power company of the DRC, Société Nationale d'Electricité (SNEL), using two transmission lines from Lubumbashi. There are pylons in place for a third line. The lines will be refurbished and re-stringed with aluminium conductors to minimise copper theft incidents.
- 12 MW of back-up power will be provided on site (new diesel gensets).
- The refurbishment of the diesel tank farm.
- Communications infrastructure required to support an operating mine.
- Leased and refurbished accommodation in Kipushi for owner's team personnel.
- A new overland conveyer for transporting ore and waste from Shaft 5, to the new plant/ore stockpile and temporary waste storage area, respectively.
- A run-of-mine ore stockpile and a temporary waste stockpile area.

- A new processing plant and supporting surface infrastructure that incorporates the following unit operations:
  - Crushing and screening.
  - Dense media separation (DMS) to remove dolomitic wastes for backfill.
  - Milling.
  - Two stage differential flotation.
  - Concentrate bagging facility.
- A new tailings dam with an overhead line supplying power to the facility.
- A new on-mine rail loading platform and the refurbished Kipushi Station and Kipushi to Munama rail spur (owned by SNCC).
- Old (refurbished) and new facilities including:
  - General office, technical buildings and structures.
  - Mine services buildings (change rooms, mess, kitchen, laundry).
  - Workshops, stores and construction laydown areas.
  - General electrical buildings.
  - Security and emergency services buildings.

Figure 18.1 Overall Proposed Site Layout

Figure by Ivanhoe, 2017.

18.2 Access Infrastructure

Apart from the specific mining and plant site areas, the interconnection upgrades are limited. The pipe rack will be replaced with a combined conveyor/pipe rack within a fenced servitude. All roads to and within the Project area are of black top construction. Therefore, with the exception of the TSF access road and rail spur refurbishment, no new access infrastructure or upgrades are required for the project. Regional access to Kipushi and local access to the mine is illustrated in Figure 18.2 and Figure 18.3 respectively.

Figure 18.2 Local Access

Figure by MDM, 2017.

Figure 18.3 Mine Access

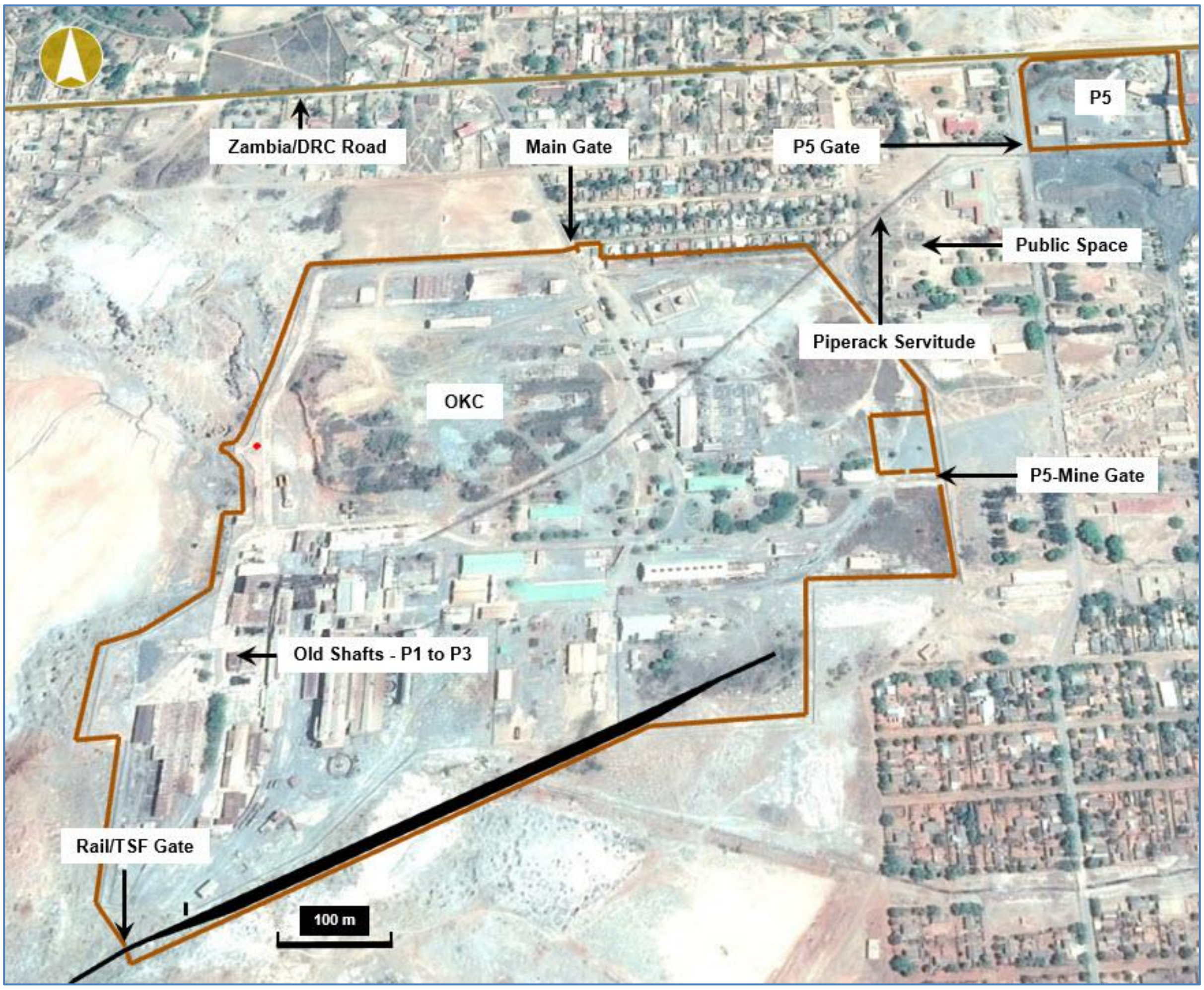


Figure by MDM, 2017.

18.3 Earthworks

Earthworks and terracing requirements were based on an engineering geotechnical investigation undertaken by SRK. The work included the excavation of 24 test pits at various areas of the mine as well as Drop Cone Penetrometer (DCP) testing adjacent to test pit positions. Selected soil samples retrieved from the test pits were submitted to Geostrada soils testing laboratory in South Africa for testing.

The geotechnical investigation showed that:

- The in-situ materials generally classify as soft materials down to a depth of three metres.
- The colluvial and residual clayey and silty soils across the site are not suitable for engineered fills, but are suitable for bulk fills.
- The high clay content of the soils will result in trafficability problems after rainfall.
- Where the soils exhibit voiding by termites, they must be treated to remove the collapse potential prior to construction.
- For individual structures with bearing pressures less than 100 kpa, deep strip foundations or engineered soil raft construction are recommended.
- For structures with bearing pressures of 100 kpa to 300 kpa, reinforced concrete rafts, spread footings or pads constructed over engineered fill are recommended.
- As hard material generally occurs below three metres depth, no allowance has been made for removal of rock.

18.4 Roads

New roads required for the project are designed to link up to the existing roads on the mine and will all be finished with a gravel wearing course. Engineering for road earthworks is subject to the same geotechnical considerations as those stated for terracing.

18.5 Weighbridge

A truck weighbridge has been provided to measure the gross and empty weights of trucks entering and leaving site. The weighbridge is 4.5 m wide and 25.65 m long and will accommodate a truck with a Gross Vehicle Mass of 80 t.

18.6 Mobile Equipment

The plant and infrastructure mobile equipment list was developed to meet project requirements. The mobile equipment presented relates to equipment provided by KICO SA for plant and infrastructure operations, including third party service providers such as: the laboratory contractor; the cleaning, catering and laundry contractor; and the security service contractor. The TSF and backfill operations will be outsourced.

18.7 Electrical Infrastructure

18.7.1 Electrical Power Supply and Switch Yard

Power is supplied by Société nationale d'électricité (SNEL) in the DRC. The Kipushi Mine, is connected to the national electrical grid through two power lines, one at 110 kV and the second at 50 kV. Both power lines are equipped with copper conductors and exposed to frequent incidents of conductors' theft. A third power line built in the nineties was vandalised. To mitigate the risk of flooding the mine in case of a prolonged power supply interruption, a project aimed at repairing this line and stringing it with aluminium conductors has been initiated. The scope will also cover the replacement of copper conductors on the existing 110 kV with aluminium conductors and modernising the equipment at both Lubumbashi RS and Kipushi terminal substations.

The incoming lines feed three transformer bays (110 x 2 and 50 kV x 1)/ 6.6 kV), adjacent to the outdoor yard. All three transformers can operate in parallel. This switchyard is in a reasonable condition and does not require any upgrade for the project.

The incoming lines feed three transformer bays (110 x 2 and 50 kV x 1)/ 6.6 kV), adjacent to the outdoor yard. All three transformers can operate in parallel. This switchyard is in a reasonable condition and does not require any upgrade for the project.

There are three existing substations:

- Main Substation adjoining the existing switchyard and transformer facility.
- Shaft 5 Substation at the shaft.
- Cascades Substation provides power to Shafts 1 to 4.

All of these substations will continue to be used and costs have been allowed for refurbishment.

18.7.2 Transmission and Distribution

The mine has an extensive system of underground tunnels that are used to distribute power from the main switchyard to Shaft 5 and to the OKC. These tunnels are all operable and will continue to be utilised going forward.

A 1.5 km, 6.6 kV overhead line will be installed to provide power to the TSF. The overhead line will incorporate an optical ground wire. This wire serves to provide both grounding and communications function between the TSF and the plant.

18.7.3 Security of Supply and Emergency Power

An outage schedule for 2016 was provided by KICO. For the year in question, power availability was high at 99.58% (37 hours of down time for year). In moving forward, the following points should be noted:

- The Kipushi mine has 4 MW of installed back-up power, supplied by two diesel generators (1 x 1 MW and 1 x 3 MW). This generation capacity was not designed to run either the mine, plant or the dewatering systems independently from the grid. It will however, run ventilation fans and the shaft hoist in an emergency.
- New 12 MW of back-up power in the form of diesel generators will be provided on site, to enable critical operations such as underground pumping, to continue in times of power outages.

18.7.4 6.6 kV Switchboard

The 6.6 kV double busbar switchboard will be retrofitted with new vacuum circuit breakers and new protection relays. This upgrade will consist of 12 new incomer circuits, 22 feeder circuits, two bus-couplers and four busbar mounted voltage transformers (VTs). The upgrade will also include an arc flash protection system and a remote switching panel. Two new battery tripping units have been allowed for. The retrofitting of new breakers and protection relays into the existing switchgear cubicles will prevent having to re-terminate any of the old power cables for the existing plant.

18.7.5 Power Factor Correction

Power Factor Correction (PFC) has been allowed for at the 6.6 kV switchboard level. The PFC design has been based on a total load of 22.7 MVA at a PF of 0.84. The power factor will be corrected to 0.93 lagging, in accordance with SNEL tariff penalty requirements. The system will consist of an outdoor enclosed PFC plant with two 2.2 MVAr steps with a 5^{th} harmonic filter. It has been assumed that all electrical loads will be connected to the same 6.6 kV busbar.

18.7.6 Low Voltage Reticulation

Secondary distribution is at 525 V. Star points of the distribution transformers will be resistively connected to earth. Transformer secondaries shall be rated at 550 V to allow for volt drop at full load.

18.7.7 MCC Substation Containers

All plant motor control centres (MCCs) will be housed in dedicated low voltage steel substations that are converted 12 m high-cube shipping containers. The containers have been insulated and fitted with a tropical roof. The substations are elevated for cable access.

18.7.8 Transformers

Electrical loads are allocated to MCCs, and associated transformers. These loads are grouped by process areas as far as practicable, considering transformer loading and voltage regulation. The MCC designs have been based on 2,500 kVA transformers. Distribution transformers are 6,600/550 V, vector group Dyn11.

Infrastructure lighting and small power would be fed from 6600 / 420 V mini-substations, while plant lighting and small power will be fed from dedicated 525/420 V transformers.

A transformer loading schedule has been completed for each transformer.

18.7.9 Motor Control Centres (MCC)

The MCCs will be of steel construction, free standing, bottom cable entry, front access and operation, fully compartmentalised design (form 3b and 4a). The operating voltage will be 525 V, 50 Hz with a control voltage of 110 V, 50 Hz supplied from an internal control transformer. The design fault level will be 50 kA at 525 V for all transformer fed MCC's. A power metre will be provided per MCC incomer and connected to the plant supervisory network.

Motor starters will be direct-on-line (DOL) where motor kW is less than or equal to 90 kW, unless otherwise specified by process requirements. Motors above 90 kW will be started by Soft Starter or Variable Speed Drive (VSD) depending on the application. DOL starters are typically equipped with a triple pole, Molded Case Circuit Breakers (MCCB), contactor (Type 2 coordination) and intelligent overload relay(s) (Simocode Pro S). Contactors will be rated for AC-3 duty.

All starter / variable speed drive (VSD) related information will be communicated over the Profibus-DP network to the PLC / SCADA system.

18.7.10 Field Equipment

All drives will be equipped with local start-stop stations with latching e-stop. These will be field mounted within robust steel drip covers. Plant start-up sirens will provide a warning for conveyor and large equipment drives about to start. All emergency functions such as emergency stops are to be hard wired, but will also be monitored by the PLC.

Generally, VSDs will be mounted within the MCC, large VSDs, mounted external to the MCC as standalone cubicles.

18.7.11 Motors

Low voltage (525 V) motors will be designed to IEC 60034, for continuous duty class S1. Insulation will be Class H. Temperature rise will be limited to 80°C (Class B). Enclosures will be IP55 to IEC 60034-5. Premium efficiency (IE3) motors will be used throughout the design and shall be of the totally enclosed fan cooling type (TEFC).

6.6 kV Motors will include Zorc Surge suppressors that will be fitted separate to the motor cable box (MV motors). Temperature monitoring devices are to be fitted to bearings and windings on MV motors.

18.7.12 Cable

Medium voltage (6.6 kV) cables will be individually screened copper conductor three core XLPE/PVC/SWA/PVC 6.35/11 kV cable to IEC 60502. Single core cables will be of XLPE/PVC/SWA/PVC construction and be arranged in prescribed trefoil formation (gland plates will be of non-ferrous construction).

Low voltage cables will be copper conductor PVC/PVC/SWA/PVC 600–1,000 V cable to IEC 60502. Standard flame-retardant cable is to be utilised for surface installations. Power cables shall have four cores, the fourth core being utilised as an effective earth between the equipment (e.g. motor) and the substation earth bar.

Conductor sizes for 525 V motor feeders shall be sized to ensure reliable motor starting. Cables are sized for a maximum 5% voltage drop during full load condition. Start-up voltage drops are determined on a case by case basis based on the starting torque requirements.

18.7.13 Cable Racks

The preference is for cables to be mounted either in the underground cable tunnels on site, or above ground level on suitable cable racks or overhead line systems. Where necessary, buried cables will be in trenches and will be provided with cable markers on surface at 10 m intervals and changes in direction as per electrical installation specification. Cable trenches will be backfilled with a suitable material to ensure effective heat transfer from the cable to the surrounding earth. A detailed services servitude plan is to be made and kept up to date.

18.7.14 Earthing

Detailed earthing and lightning protection design will be carried out for new areas. The earthing values shall be in accordance with IEC 62305 Parts 1, 3, and 4.

Earth reading values of less than 10 ohm shall apply for general plant structures and conveyors. All electrical equipment shall be earth bonded via the substation earth bar.

Lightning masts, shall be earthed using copper rods to give an earth resistivity of a maximum of 5 ohm. The various earths shall be linked using buried 70 mm^2 bare copper earth wire.

18.7.15 Lighting and Small Power

The lighting and small power design shall generally comply with the provisions of IEC 60364. The additional lighting and small power for the project will be an extension of the current systems. Lighting will be by a combination of fluorescent, bulkhead and floodlights to achieve illumination levels required. The lighting levels are detailed in the Electrical Design Criteria (EDC).

Emergency lighting has been allowed for in key areas. Emergency lights will be fed from dedicated UPS circuits. Photoelectric switches will control the exterior lights. Provision has been made for weatherproof 230 V 16 A switched socket outlets and 525 V 63 A welding socket outlets.

General area 25 m lighting masts will be provided for the plant terrace, the waste stockpile terrace, the ROM stockpile terrace, the TSF and for parking areas.

18.8 Water Management, Supply and Distribution

18.8.1 Overview

There are four primary sources of water on site, namely:

- Town / potable water from a spring.
- Underground water from Shaft 5.
- Underground water from Shaft 3.
- Water pumped from the pit.

With respect to water management on site, it is relevant to note that:

- Raw water for the plant and for surface infrastructure is sourced from the underground workings.
- Water is not recovered from the tailings storage facility (TSF) or from site drains.
- Underground water, which meets the DRC discharge requirements, is discharged directly to the site stormwater drains.
- The mine largely falls in one catchment area (southern catchment area), with water running west to east and north to south-east to the Kipushi river via tailings dam 3.

18.8.2 Potable Water

Potable water for the mine and the town has been reviewed by Golder and costs have been provided to KICO for the refurbishment of the town's water supply system for incorporation in the Kipushi 2017 PFS estimate.

- Supply:
  - Number of boreholes: 10
  - Volumetric flowrate required: 5 ML/d (excluding mine requirements)
- Storage:
  - Northern water tower: 3.5 ML
  - Southern water tower: 2.0 ML
- Distribution: Gravity
- On site potable water storage (new): 50 m^3
- Mine potable water usage (proposed): 67.5 m^3/d

18.8.3 Open Pit Water

The design volumetric flowrate from the pit is 600 m^3/h (seasonal). Additional information on the pumping systems and associated requirements will be developed during the FS.

18.8.4 Shaft 3 (Cascades) Water

The design volumetric flowrate from the underground workings at P3 is 2,100 m^3/h. Additional information on the pumping systems and associated requirements will be developed during the FS.

18.8.5 Shaft 5 Water

The design volumetric flowrate from the underground workings at P5 is 2,400 m^3/h. The underground pumping system is described in the mining sections of this report.

18.8.6 Raw Water Supply to Mine

Water pumped from underground was historically pumped to the raw water tower adjacent to Shaft 5, from where it was distributed to various users, with the balance discharged to the stormwater drains. Given that there are currently no water users, and the water meets the DRC discharge requirements, the water tower has been placed on a care and maintenance basis and water is discharged directly to the various site drains.

The Kipushi 2017 PFS has allowed for:

- Refurbishing the water tower.
- Replacing all steel water pipes between Shaft 5, Shaft 3, and the water tower, with equivalent sized pipes fabricated from High Density Polyethylene (HDPE).
- Installing piping for conveyor and stockpile dust suppression systems.

User of underground water and the approximate quantity of water used are defined below:

- The plant will require 16.3 m^3/h of underground make-up water, if the water from concentrate and tails thickeners are returned to the plant; if not raw water make-up will increase to 26 m^3/h.
- Vehicle workshops (wash bays) and the fixed dust suppression system will require 10 m^3/h of underground water.
- Dust suppression requirements (mobile equipment) - to be defined in the FS.
- Fire water system (adhoc user).

Excess water from underground will be blended in with return water from the TSF after neutralisation and discharged into the existing stormwater drainage system.

18.8.7 Stormwater Management

There are two catchment areas associated with the mine, namely the northern catchment area that runs to the north of the road between Shaft 5 and Lake Kamalenge and the southern catchment area, which runs to the south of the aforesaid road and drains to the Kipushi River via TSF 3. The sites drainage system is highlighted in Figure 18.4.

For the Kipushi 2017 PFS, only the southern catchment area is used. Given that the drains within MDM's scope of work are currently being used and are in a reasonable condition, no capital cost allowance has been provided for stormwater drainage systems. Rather, an ongoing maintenance function and budget has been allowed for in the operating cost estimate for cleaning and repairs. The premise that the South catchment drain (green line) is not relevant to the projects, will be revisited as part of the FS.

Figure 18.4 Kipushi Mine Drainage System

18.8.8 Discharge of Water from Mine Site

Currently, all water falling on the site and/or emanating from underground meets the DRC Government's discharge requirements and is discharged directly into the environment without treatment or containment.

18.9 Public Health Services

There are no sewage treatment plants on site. Ablution facilities drain into dedicated septic tanks, which over flow into French drains. This practice will continue.

18.10 Fire and Emergency Services

An existing building, currently in use will be upgraded to provide fire and emergency services facilities to house emergency personnel, equipment and emergency vehicles.

A separate mines rescue room has been allowed for, to be equipped by a specialist service provider.

18.11 Fire Protection

Fixed fire-fighting infrastructure will be installed to supplement the mine's mobile fire-fighting capability. Fire water will be drawn from the plants raw water system (underground water), and allowance has been made for provision of new fire water pumps, and reticulation to both new and old buildings.

Fire-fighting apparatus such as hose reels and hydrants have been estimated for inclusion in the new and refurbished buildings. For conveyors, installation of fire hydrants, fire hose reels have been allowed for at specific distances and fire extinguishers have been provided at each drive, take-up and tail and transfer tower. Deluge systems have been allowed on equipment with hydraulic power packs.

Fire detection will be included in new LV substations, with the present MV substations using their existing fire detection systems which are assumed to be adequate. Hand held fire extinguishers will be placed in and around each new LV substation for firefighting purposes.

In addition to the fixed fire-fighting equipment provided, one fire tender and one fire-fighting land cruiser have been allowed for in the estimate. Water for the operation of this equipment will be sourced from either the fire water system or mobile dust suppression tankers (when working remotely from fixed infrastructure).

18.12 Fuel and Lubricant Supply

There is a historic diesel tank farm on site, comprising two, 750 m^3 storage tanks located within a bunded area (Figure 18.5). The tank farm supplies fuel underground (via a borehole), to the generator day tanks, to the incinerator, and to two or more local distribution tanks for vehicle fuelling. Two small trailer mounted fuel tanks have also been allowed for remote dispensing.

Figure 18.5 Tank Farm

As part of an external services contract, a fuel provider, will refurbish and operate said fuel farm over a defined contractual term. The tank farm has of the order of two months diesel storage capacity at the base projected fuel consumption rate.

Lubricants delivered, will be stored in a new 84 m^2 portal frame structure, whilst waste oils will be stored in an adjacent 84 m^2 portal frame structure. Both facilities lie on a concrete bunded slab.

18.13 Communications and IT

Currently on site, KICO communicates internally and externally using:

- A satellite connection (c-band).
- Combined fibre (5 MBps) and line of site radio connection mounted on the Shaft 5 tower.
- Cellular connection (Vodacom).
- Television (DSTV or equivalent connection).
- Radio.
- Wireless connection on site and in the guesthouses.

An allowance has been made for refurbishing the existing telephones and servers on site. This facility will host the relevant computer servers and the PABX system. The existing telephone cabling system that connected the exchange, with the various buildings on site (offices, stores and workshops), will be replaced with a fibre optic network and within the buildings, Cat 6 cabling will be installed for the provision of data and voice services.

Addition points to note:

- Plant/process control data, security and general communication system data will be run on separate networks.
- The TSF and the main site will be connected using an Optical Ground Wire (OPWG) on the overhead power line.
- The wireless system will be upgraded to provide reliable services across the entire site.

IT hardware and software (including specialist software) for has been allowed for in each of the areas and departments.

18.14 Waste Management

The disposal requirements will be undertaken in accordance with the project environmental management plan (EMP). The waste management facilities will include:

- Storage of Waste Petroleum Products.
- Storage of Scrap and/or Recyclable Products.
- Off-site Septic Tank Sludge Disposal.
- Storage of Waste Prior to Landfill.
- Off-site Landfill Disposal.
- Incinerator.

18.15 Buildings / Structures

As far as practically possible, old buildings and structure will be refurbished and, in some cases, repurposed to meet project requirements. Where necessary, new buildings and structures have been allowed for. Given the relatively short mine life, the approach has been to ensure that building refurbishment is fit for purpose and new buildings are either of the portal frame, prefabricated or containerised types. These are shown in Table 18.1.

Table 18.1 Kipushi 2017 PFS Building, Structures and Rooms

Administration building	Crib Rooms and Toilets
Bonded Stores Offices	General Machine Workshop
Technical Services Building	Boiler Maker Workshop
Geology office	Sandblasting Workshop
New training offices	New joinery / Masonry Workshop
Control Rooms	Light/Heavy Vehicle Workshop
P5 Shift Meeting Area	Vehicle Wash Bay
Community office	Stores and Construction Laydown
Technical Buildings	Store - New Heavy Equipment Store
Generator Building	Store - New Mine Light Store
IT and Server Room	Store - Plant Bags
Core Stores	New Reagent Make-up & Storage Area
Lamp Room / Switch Room	Store - Furniture and Fixtures
New Laboratory	Store - Twin Store Building
Packaging Plant	Store - Gas Bottle
Shaft 5 Change House and Lamp Room	General Electrical Buildings (Main Substation, P5 Substation, Cascades Substation)
Mine Change Rooms	Security and Emergency Services
Kitchen	Main Site Clinic
Mess	Fire and Emergency Services Building
Laundry	Shaft 5 Mines Rescue Room

18.16 Laboratory

Whilst there is an existing partially functioning laboratory on site, comprising of a 350 m^2 analytical lab and a 320 m^2 sample preparation lab, it was decided to not refurbish these facilities on the basis that the sample preparation laboratory is not suitable for the new duty and the cost of refurbishing the existing facilities to meet the new project demands were too similar.

A new portal frame/containerised laboratory with the requisite equipment has been allowed for, as described more fully below. The initial scope of work as defined by MDM was subsequently amended to exclude the analysis of germanium, on the basis of cost.

The laboratory facility includes:

- Bulk sample preparation laboratory:
  - 27 m x 34 m concrete slab (918 m^2).
  - 10 m x 30 m portal frame structure and equipment (US$296,000).
  - Receive and process "skip bins, > 1 t sample" and/or +20 mm material.
- Analytical laboratory:
  - 21 m x 42 m concrete slab (876 m^2).
  - 6 x 40' containers, with each container serving a specialist function and a roof that spans the containers.
  - 1 x 40 - toilet / ablution facility.
- Laboratory information management system.

18.17 Workshops

Historically the site was largely self-reliant with respect for the maintenance of equipment. The business model employed was of one large central site workshop, supported by a number of smaller workshops in different geographic/business areas. It is planned to consolidate the workshops and the remote workshops be re-purposed.

The workshops planned are:

- General Machine Workshop: existing building for mechanical/machine work shop, hydraulic workshop, electrical and instrumentation workshop.
- Welding Workshop: existing building for site welding activities.
- Sandblasting: existing building currently used for sandblasting.
- Joinery and masonry workshop and store: existing building for carpentry and storage of masonry products and supplies.
- Light and Heavy Vehicle Workshop: existing building for maintenance of mobile equipment, tyre changing, fuel and lubrication, included in the refit will be new canage.
- Vehicle wash down bay: a new facility of packaged washing equipment, tanks and pumps including waste management controls.

18.18 Stores and Construction Laydown

The store buildings and laydown areas will be a mix of new and existing facilities. The key facilities are:

- Heavy Equipment Store: a new facility using an existing building.
- Mine Light Equipment Store: a new facility using an existing building with laydown area.
- Flammable Stores: a new facility using an existing building. A sperate store is planned for gas bottles.
- Concentrate Bags Store: a new facility using an existing building.
- Reagent Make-up and Storage Area: a new building.
- Building Maintenance Stores: an existing building for building supplies.
- Bonded Stores and Offices: existing facilities are to be moved to other existing buildings.
- Shaft Cable Store: existing building.
- Construction Store Building: an existing building for construction contractors
- Laydown Areas: continue with existing areas, new gate houses will be installed.

18.19 Security and Access Control

The main access control points for the site are:

- Main gate (northern entrance).
- Shaft 5 gate house (western entrance).
- Main Mine-Shaft 5 gate house (eastern entrance).
- Shaft 5 main road gate (northern entrance).
- Rail and TSF road gates (southern entrance).
- TSF gates.

The fencing is summarised below:

- Type (existing): ClearVu (www.clearvu.com), 3 m high with spikes.
- Total fence line: 8,211 m.
- Shaft 5 and Plant - New fencing required: 2,897 m.
- Fencing to be taken down, moved and re-instated: 700 m.
- TSF fencing: 1,840 m.

Cameras have been allowed for on the Kipushi Mine connected to the security workstations via the sites fibre optic network. Access control systems comprising of manual and automated gates, turn-styles, personnel readers and linked CCTV systems have been provided for in the cost estimate, along with the associated time and attendance software, badges and cards.

18.20 Accommodation

Accommodation is provided in the town of Kipushi to personnel and to some mine subcontractors. No housing or offices have been allowed for in Lubumbashi. Accommodation costs are planned to be included in the contractor costs. No accommodation has been provided for construction personnel, on the basis that:

- The appointed earthworks, civils and SMPP contractors will be local to Lubumbashi.
- Personnel associated with vendors and the E&I and EPCM contractor, will be accommodated in hotels or guest houses locally.

The company does not own accommodation in Kipushi, but rather refurbishes and leases houses in the town.

For the delivery of accommodation services, a third-party service provider will be employed to provide a centralised mess at the main guest house and a laundry function at the mine.

18.21 Concentrate Transport and Logistics

18.21.1 Transport and Logistics Summary

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

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

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 Munama and through to the Zambian boarder at Ndola.

On 30 October 2017, Ivanhoe Mines and the DRC's state-owned railway company, Société Nationale des Chemins de Fer du Congo (SNCC), signed a MOU to rebuild 34 kilometres of track to connect the Kipushi Mine with the DRC national railway at Munama, south of the mining capital of Lubumbashi.

Under the terms of the MOU, Ivanhoe has appointed R&H Rail (Pty) Ltd. to conduct a front-end engineering design study to assess the scope and cost of rebuilding the spur line from the Kipushi Mine to the main Lubumbashi-Sakania railway at Munama. The study has begun and construction on the Kipushi-Munama spur line could start in late 2018. Ivanhoe will finance the estimated US$32 million (plus contingency) capital cost for the rebuilding, which is included within the overall Kipushi 2017 PFS capital cost.

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 North–South Rail Corridor to Durban via Zimbabwe is fully operational and has significant excess capacity.

For the direct rail option, the development of a rail loading facility at the mine and the rebuilding of the 34 km rail track between Kipushi and Munama, where it links up with the existing North–South Corridor, will be required. Trains operated by SNCC can then be brought to the mine for loading, and customs clearing can be done at the mine, before railing to the export ocean port, shown in Figure 18.6. It is estimated that the rebuilding of the Kipushi to Munama railway line will take 23 months.

The existing Kipushi Station will 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.

The rail operator would need to source this fleet of rolling stock and establish a dedicated pool of wagons to service Kipushi. This equipment could either be sourced new from an overseas manufacturer (India or China) or be provided by establishing a PSP with Transnet to purchase and rehabilitate a portion of their existing 'B' fleet wagons.

The study has assumed a combination of containerised and break bulk concentrate out of Durban to China (Shanghai).

Figure 18.6 DRC to South Africa North–South Rail Corridor

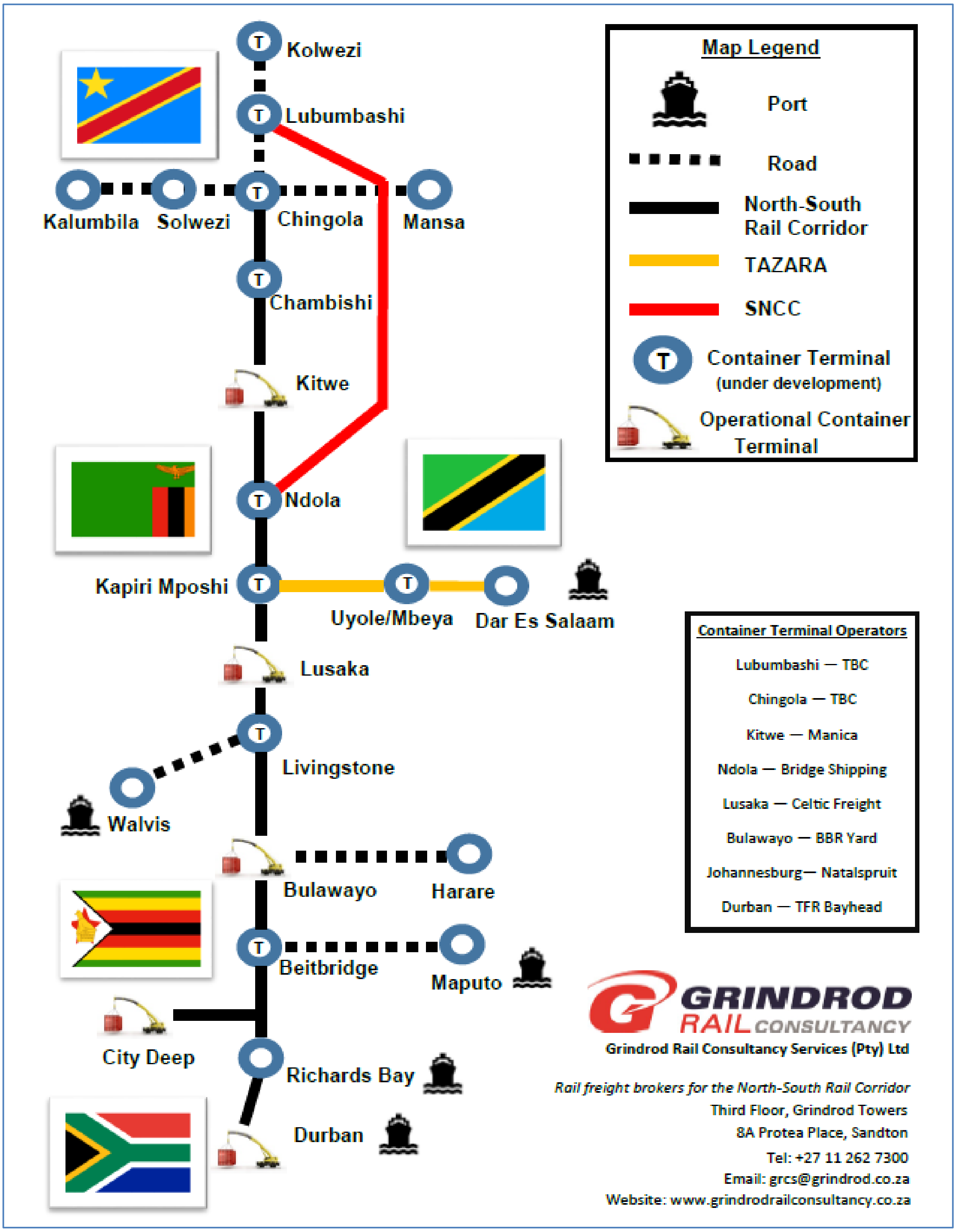


Figure by Grindrod, 2016.

18.21.2 Transport Options

Both rail and road concentrate transport options were reviewed. There are various established road corridors from Kipushi to an ocean sea port within the Southern Africa Development Community (SADC) region. All of these routes are supported and promoted by the SADC.

The base case for the PFS is a direct rail option, including the development of a rail loading facility at the mine, and the rebuilding of the 34 km rail track between Kipushi and Munama, where it links up with the existing North–South Corridor. This corridor is currently operating on a daily basis, and trains are loading regularly as far north as Likasi and Kolwezi for haulage to Durban. The operational risks are normal derailments (rare) and performance of the three parastatal rail operators on the corridor: SNCC, Zambia Railways (ZRL) and Transnet Freight Rail (TFR). The major infrastructure risk on the North–South Rail corridor is the track itself and possibly the Victoria Falls bridge.

The principal risk for the road haul options from Kipushi to Impala Terminals' intermodal facility on the Likasi Road is that the proposed Impala loading facility on the Likasi Road north of Lubumbashi is currently only partly constructed and has been mothballed pending new investment.

The road haul from Kipushi to Ndola via Solwezi using the Kipushi border post, followed by rail haulage from Ndola to Durban/Richards Bay assumes that re-opening and upgrading of the Kipushi border post between the DRC and Zambia can be negotiated at a government level in Kinshasa.

Direct road haulage from Kipushi to Walvis Bay via Solwezi and the Kipushi border post would more than likely be subject to severe constraints for this route if an additional 50 trucks per day were added to current traffic volumes. The road conditions in Zambia from Solwezi through to the Caprivi strip in Nambia are also far from ideal, with some sections along this route in poor condition and would probably be subject to delays during the rainy season.

18.21.3 Direct Rail Transport

The direct rail option will require the refurbishment of the Kipushi Station infrastructure, a rail loading facility and the rebuilding of the 34 km rail track between Kipushi and Munama, where it links up with the existing North–South Corridor. Trains operated by SNCC can then be brought to the mine for loading and customs clearing can be done at the mine, before railing to the export ocean port.

18.21.4 Kipushi Station Rail Infrastructure

The existing Kipushi Station will 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. The Kipushi station is proposed to be partially rebuilt to serve as a customs clearing and staging area. The scope of the rebuild includes:

- The provision of two staging lines in loop formation, each capable of staging a 25 wagon trains (410 m between clearance markers, also allowing for two locomotives) as well as a main/through line.
- All existing track and turnouts to be uplifted, stacked and handed over to the owner of the material which is understood to be Gécamines.
- Existing ballast to be cleaned and reused as bulk fill material.
- Based on the geotechnical report the existing formation layers in the station have below standard CBR values. Therefore, the formation roadbed shall be rip and compacted and new formation layers shall be constructed.
- New track shall be constructed using 40 kg/m rail, steel sleepers at 700 mm centres and ballast distributed at 1,000 m^3/km.
- The final track levels shall be constructed at a grade of less than 1:800, which is the standard for staging areas.
- Area lighting to be provided to allow for a 24-hour operation.
- The existing station buildings will be used by customs officials with nominal allowance made for minor refurbishment.
- The rail loading facility will also cater for the loading of road vehicles including a weighbridge, at no significant extra cost.

18.21.5 Current Rail Corridors

The railway connection from South Africa to the Copperbelt is today vastly underutilised, with carried annual transit freight volumes of only 288 kt in 2016 compared to its current capacity of around 3 Mt. The full rail infrastructure route from Kipushi to Durban is operational and any problems arising from sections of track in poor condition are overcome by running trains at slower speeds. These slower speeds are offset by night operation of trains (whereas many road trucks cannot move in darkness) and the much faster clearance of rail wagons at international borders (two hours in most cases as goods travel in bond). The resultant average speed of a train on the North–South Rail Corridor (NSRC) in 2016 was about 16 kph. In comparison to road, with night travel by rail and minimised border delays, the journey from Lubumbashi to Durban can be achieved by rail in 200 hours, or nine days which is as fast as currently achieved by an average road convoy. The SNCC rail network in the Haut-Katanga Province is shown in Figure 18.7.

In the first quarter of 2017, the North–South Rail Corridor operated approximately thirty trains every day along the full length of the corridor. Approximately 90% of the corridor's capacity is currently unused. KICO would require under two trains per day from Kipushi.

Figure 18.7 SNCC Katanga Rail Network

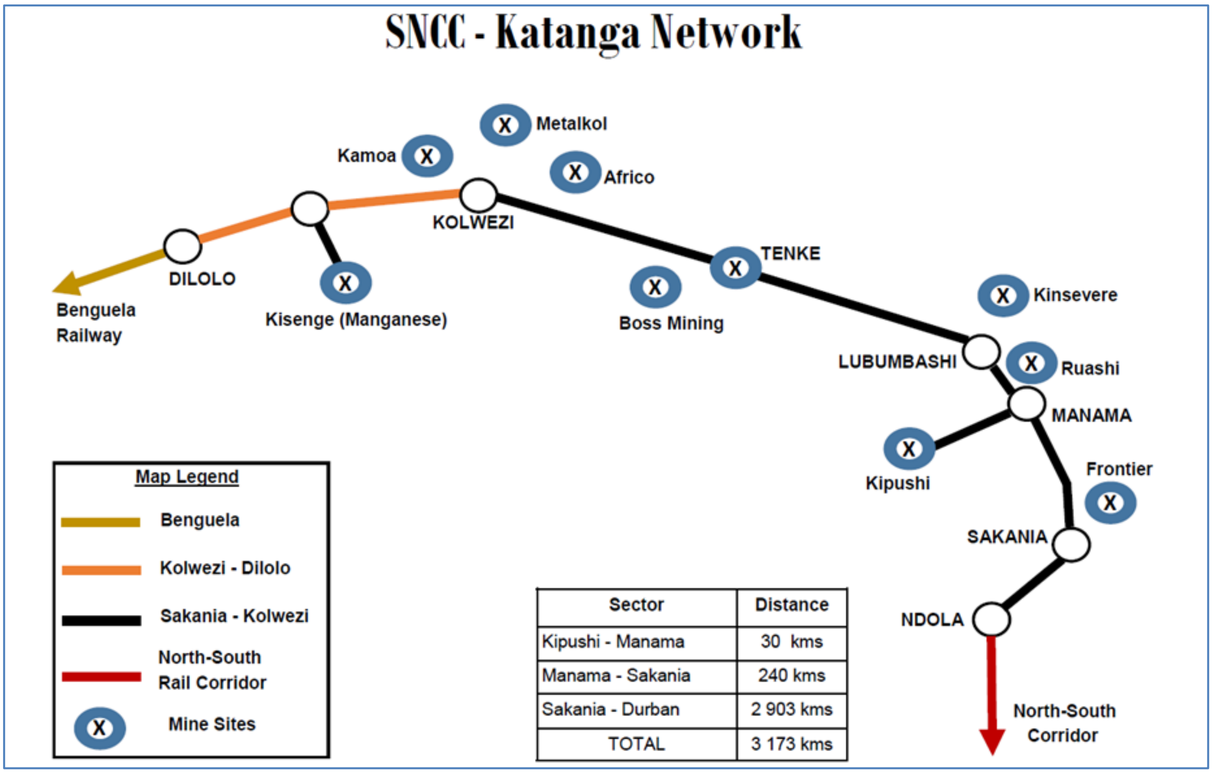


Figure by Grindrod, 2016.

Table 18.2 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**

Figure 18.8 SADC Rail Network

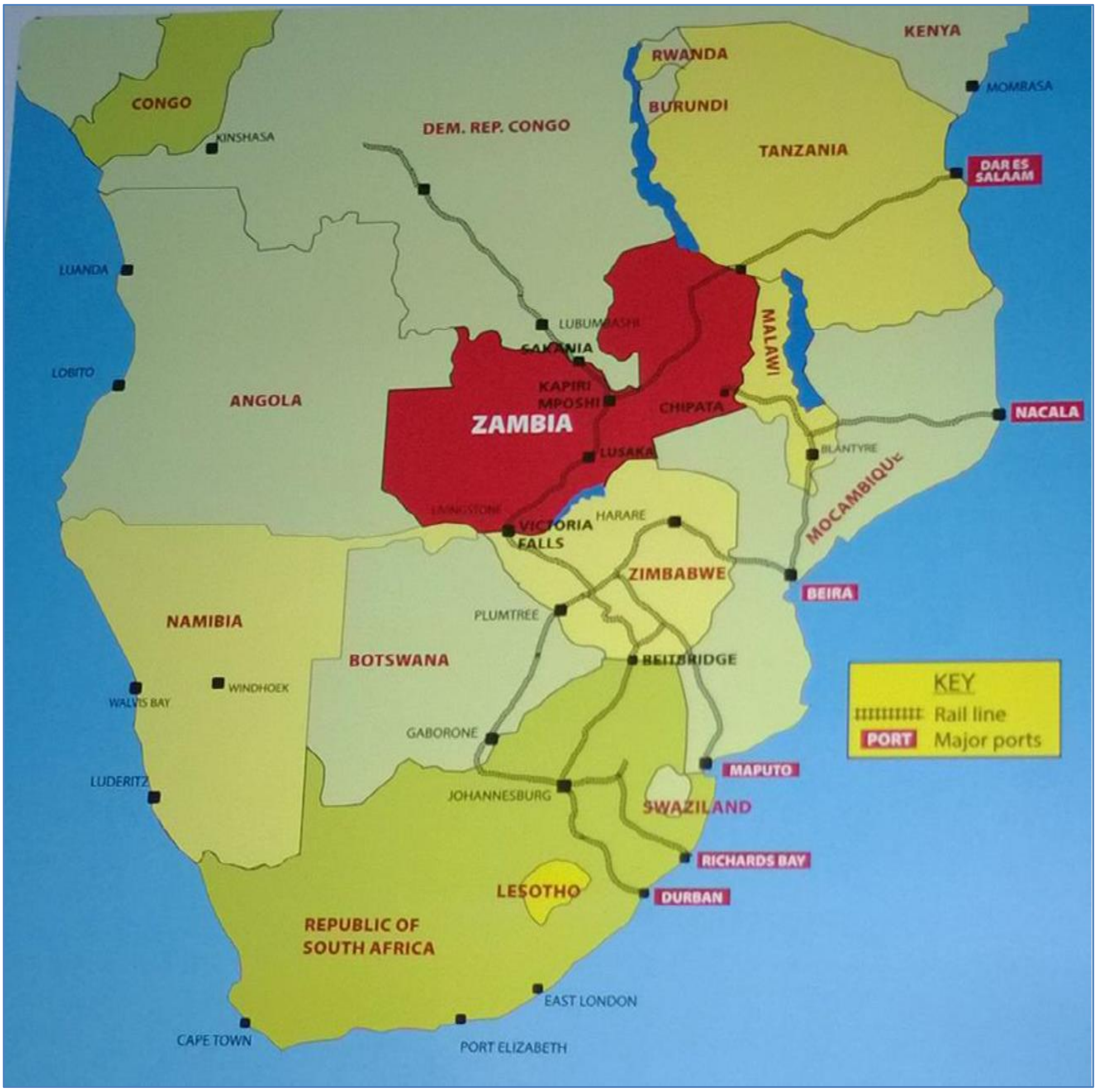


Figure by Grindrod, 2016.

18.21.6 Kipushi to Munama Rail Line

The condition of the line was inspected for the PFS and a report was prepared by Grinrod. Key points relating to the condition are:

- The majority of rail and sleepers have been removed. Where rail and sleepers have not been removed, it was found to be severely corroded and damaged and not suitable for re-use.
- Ballast originally used was largely in place, but contaminated and degraded to the extent that it is not suitable for re-use as ballast.
- The sub-ballast formation was largely intact, although damaged by erosion in isolated areas.
- Drainage channels and structures were damaged in isolated areas and regarded as insufficient leading to the erosion seen during the line assessment.
- The formation geometry (vertical alignment/curve radii and horizontal alignment/gradients) is regarded as acceptable.

The work to rebuild the rail line and station was identified as:

- Remove and hand over to the existing owner (SNCC) all remaining rail and sleepers.
- Repair damage to existing formation in isolated areas. Existing ballast and imported fill is to be used for this purpose.
- Repair existing drainage structures and re-shape drainage channels and berms to ensure proper storm-water drainage and protection of the formation against erosion.
- Rip and re-compact top of formation using existing ballast to increase the bearing capacity of the top layer.
- Import new ballast at 1 m^3/m and install 40 kg/m rails on steel sleepers at 700 mm spacing.

18.21.7 Design and Construction Schedule

It is estimated that the rebuilding of the Kipushi to Munama railway line will take 23 months, inclusive of detail design, tendering to appoint a construction contractor and construction. The design period of 8 months can overlap with negotiations, but construction can only commence once the agreement has been concluded.

18.21.8 Rail Operation Plan

For the Kipushi to Richards Bay rail journey, a transit time of 10 days is anticipated broken down by sector shown in Figure 18.9.

Figure 18.9 Train Service Chart – Kipushi to Durban

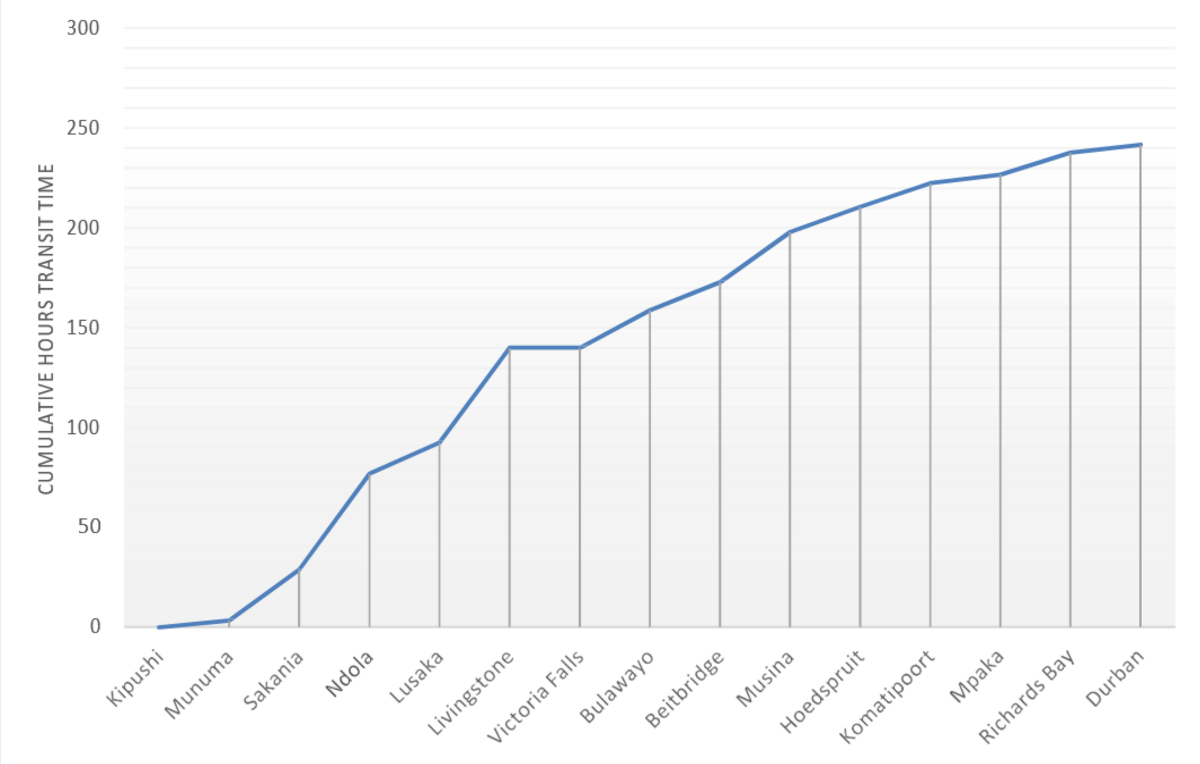


Figure by Grindrod, 2017.

After all Customs clearance procedures are finalised at Kipushi station, loaded wagons would depart in blocks of 25 box cars with an estimated payload of 1,000 t per rake. These blocks would be hauled from Kipushi to Sakania at the DRC border with Zambia by one locomotive. Although the minimum passing loop length between Lubumbashi and Sakania is 410 m allowing for trains of up to 25 wagons, SNCC in the past generally restricted train lengths to a maximum of 15 wagons due to the limited traction power of locomotives.

The rail operator would need to source this fleet of rolling stock and establish a dedicated pool of wagons to service Kipushi. This equipment could either be sourced new from an overseas supplier or be provided by establishing a PSP with Transnet to purchase and rehabilitate a portion of their existing 'B' fleet wagons.

18.21.9 Ocean Shipping/Freight Rates

The Kipushi 2017 PFS considered the merits and disadvantages of transporting zinc concentrate from Kipushi in bulk and bagged modes or containerising inland at the mine. The concentrate is then to be shipped out of Durban to China (Shanghai).

The analysis shows that a break bulk solution would be the most cost effective, whereby concentrate is prepared for shipment in 'big bags' of up to 2.2 t each at the mine, and hauled on rail in open box wagons to a terminal near Durban or in Richards Bay.

For shipment parcel sizes of up to 5,000 t, bags would be packed ten to a box inside a standard 20-foot shipping container. This containerised solution would allow the project to take advantage of cheap backhaul container shipping rates out of Durban to the Far East.

18.21.10 Existing Road Transport Corridors

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

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

It has been reported that substantial progress has been made in customs processes at international borders, as road haulage freight has increased, most main road arteries in the region are seriously congested, and traffic at border crossings often takes days rather than hours to clear. Figure 18.10, Figure 18.11, Figure 18.12, Figure 18.13, and Figure 18.14 show the following road routes from Kipushi to various ports:

- Kipushi to Durban via Road (2,716 km, 3 border crossings).
- Kipushi to Richards Bay via Road (2,604 km, 3 border crossings).
- Kipushi to Maputo via Road (2,300 km, 4 border crossings).
- Kipushi to Beira via Road (1,605 km, 3 border crossings).
- Kipushi to Dar Es Salaam via Road (2,039 km, 2 border crossings).

Figure 18.10 Kipushi to Durban via Road

Figure by Grindrod, 2017.

Figure 18.11 Kipushi to Richards Bay via Road

Figure by Grindrod, 2017.

Figure 18.12 Kipushi to Maputo via Road

Figure by Grindrod, 2017.

Figure 18.13 Kipushi to Beira via Road

Figure by Grindrod, 2017.

Figure 18.14 Kipushi to Dar Es Salaam via Road

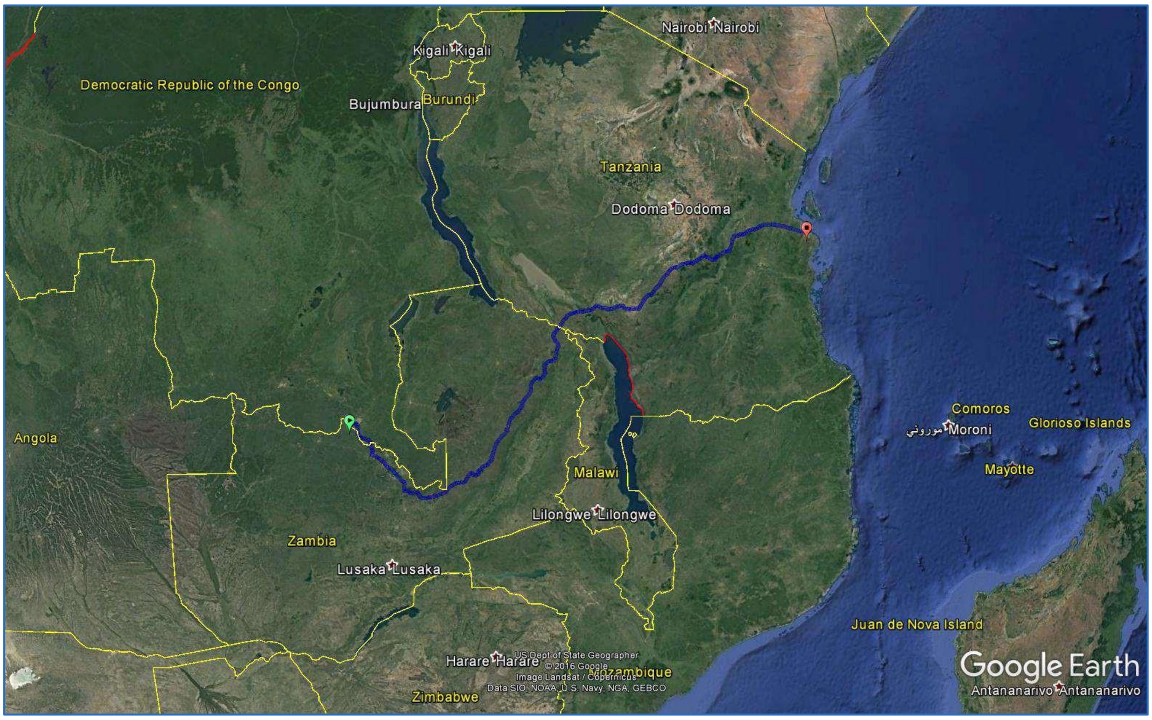


Figure by Grindrod, 2017.

18.21.11 Insurance and Inspection

Ocean marine cargo insurance can be obtained for all concentrates shipped by vessel. Under CIF contracts, marine insurance is taken out by the seller in the name of the buyer in the amount of 110% of the estimated value of the concentrates in each shipment. Risk of loss, excluding normal handling losses, passes to the buyer as concentrates are progressively loaded onto the carrying vessel. Marine insurance rates typically average around 0.05%–0.07% of the estimated invoice value (adjusted to 110%), i.e. the payable metal value, less all treatment and refining charges, as well as any penalties and price participation which may apply (the Net Invoice Value, or NIV).

Inspection services are typically employed at the vessel discharge and at the weighing and sampling procedures to ensure that the Seller's interests with respect to the proper handling of the concentrates at the receiver's facilities are fully respected. There are a number of companies that offer these services.

Where a company representative cannot be available to observe vessel loading (and/or conduct regular site visits to ensure the concentrate is being properly stored and handled) shipper's will frequently have representation at the loadport to monitor terminal activities.

18.21.12 Comments on Section 18

The mine infrastructure has a number of challenges due to the historic established mine and the subsequent town expansions in proximity to it. Notwithstanding, successful reinstatement of operations can be achieved, but will require careful management during construction and operations.

The onsite rail, road piping and electrical routes still exist but have suffered significantly from neglect and will require extensive repairs. Old onsite buildings and structures, will require moderate to extensive repair to meet the project requirements.

Overall, infrastructure has a large scope and potential risk, but an extensive review in this study has reduced these to give a good understanding of the potential work required and the associated costs to ensure the mine can function effectively.

The mine is producing a high volume of zinc concentrate product, which requires good transport infrastructure to export from site, as well the associated importation of various reagents and materials for the process. This covers rail transport from site, intercountry and through ports. Further work is required in this area to re-establish final rail routes to the mine, to link into the good condition intercountry rail routes. Transporting to South Africa for port loading and subsequent shipping, are not identified as a risk.

19 MARKET STUDIES AND CONTRACTS

This section has not been changed from the Kipushi 2017 Prefeasibility Study and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Kipushi 2017 PFS.

19.1 Zinc Market Overview

The Kipushi 2017 PFS plans for the sale of zinc concentrate. KICO have undertaken market analysis and engaged with potential customers for the Kipushi zinc concentrate. The conclusions from this work are that the Kipushi zinc concentrate will be saleable into the global zinc market. The global demand for refined zinc (Table 19.1) has grown by close to 2.5 Mt over the past decade. As with most other metals, China has become the largest participant 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 with growth at 2%–3% in the medium term. The key risk to this outlook remains the strength of global economic growth, and Chinese economic growth in particular.

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.
- The quasi-regular appearance on the exchanges of large quantities of unreported stocks.

Table 19.1 Global Refined Zinc Supply-Demand Balance

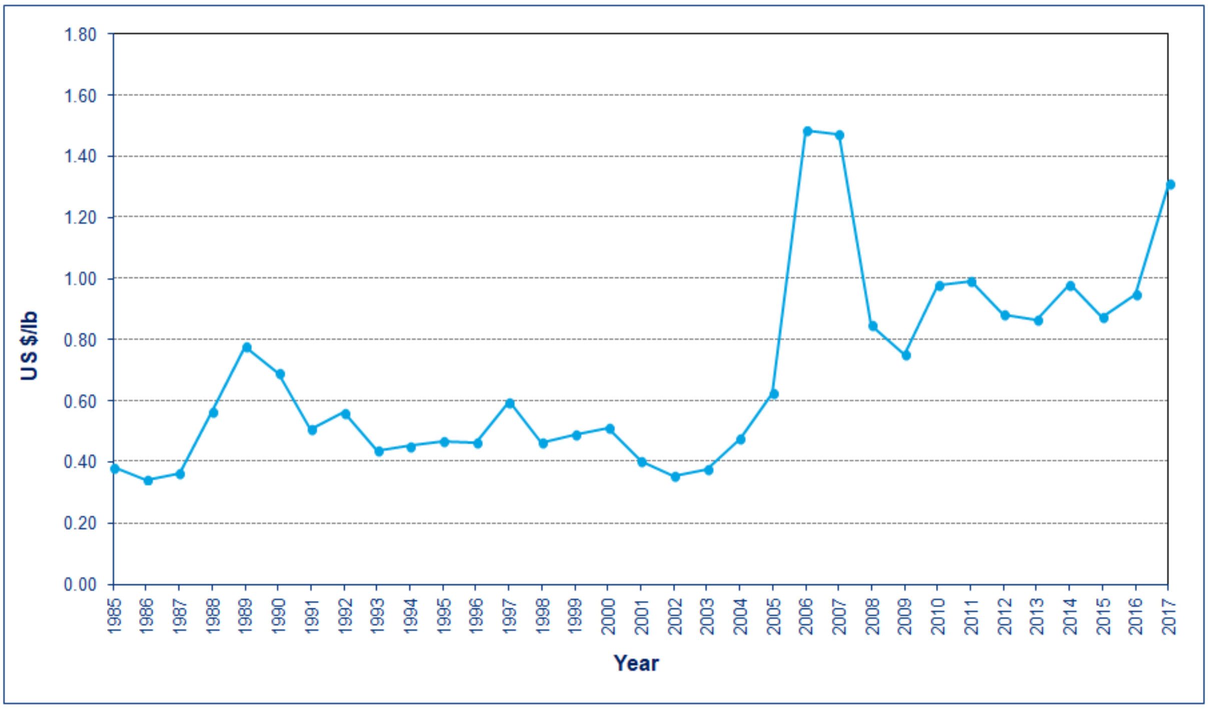
		2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019
Zn Supply	kt	11,280	12,896	13,066	12,630	12,873	13,304	14,102	14,734	14,935	15,421	15,452
Global Demand	kt	10,757	12,702	12,696	12,285	12,933	13,536	14,124	14,786	15,204	15,559	15,730
China	kt	*4,500*	*5,453*	*5,458*	*5,243*	*5,703*	*6,204*	*6,662*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

Source: Wood Mackenzie, RBCCM, ILZSG. Glencore.

Over the last few years major mine closures in Australia, Ireland and North American mines have removed 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 $0.27/lb to $0.97/lb but averaged less than $0.50/lb (Figure 19.1).

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



Source: Wood Mackenzie.

A collective underinvestment in exploration and new zinc mine capacity has contributed to declining mine supply from traditional regions and the current poor development pipeline is expected to affect short, medium, and even long-term zinc supplies. The legacy of this limited investment has been few new significant zinc discoveries. Many of the projects currently in train have been known for many years but have not been developed due to their higher cost structures and/or other challenges (e.g. technical issues, political risk, or lack of infrastructure).

19.1.1 Market Factors

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

- Market Influence of China.
- Market Consolidation.

China has a significant influence on the zinc market. China is the world's largest producer of zinc; accounting for roughly 37% of global mine zinc production according to International Lead Zinc Study Group (ILZSG) statistics. The Chinese industry is dominated by a multitude of small mines, many of which are reportedly low-grade; running with head grades as low as 3% combined Zn+Pb. Due to their scale and sheer number, it is extremely difficult to quantify actual Chinese production. As the world's largest zinc concentrate producer and as a major concentrate importer, swings in Chinese mine production can significantly influence market balances. Although the pace of expansion in mine output is expected to slow, the potential for ongoing growth could impact the projected world zinc supply contraction scenario.

Urbanisation and industrialisation will remain the dominant driving force behind global zinc consumption. Although the prospects for the developing world economies have deteriorated in recent years, the unstoppable forces of urbanisation and industrialisation mean that in the long term, the developing world will continue to dominate global growth in zinc consumption.

The potential for further zinc industry consolidation may also have a bearing on future concentrate supply. An industry dominated by fewer larger players, each with multiple projects in their portfolio, may contribute to a more disciplined introduction of new mine supply or offer cuts to existing production in an effort to rebalance the market and support prices.

19.1.2 Zinc Smelter Production and Concentrate Demand

The rate of growth of global zinc refining capacity is reported to be slowing and can be attributed to many factors, including:

- Reduced profitability due to falling processing charges.
- Concerns about longer term security of concentrate supply.
- Stagnant growth in local metal consumption.
- Rising energy costs.
- Higher capital cost requirements.
- Increasing environmental and social challenges.

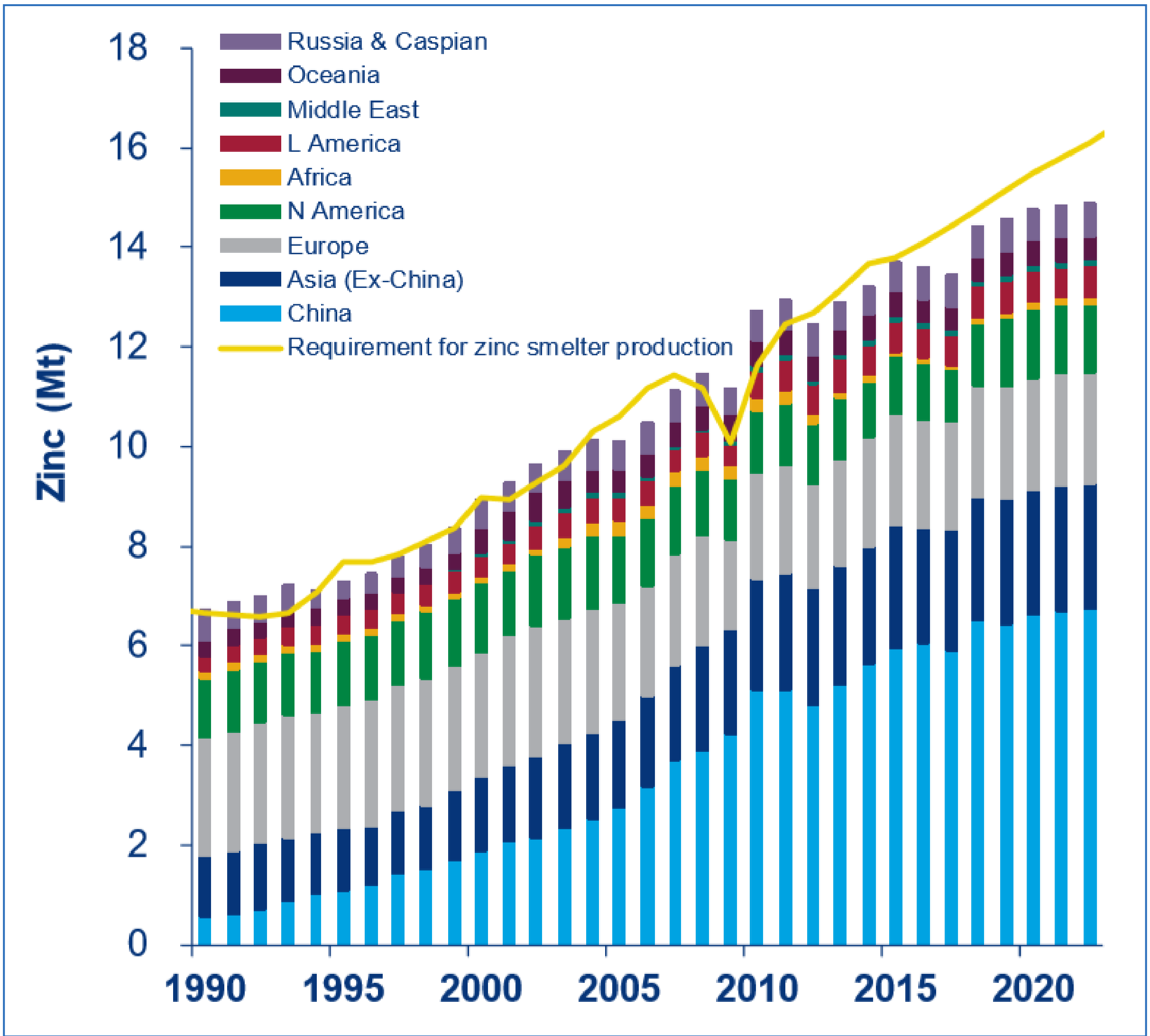
Global refined production however is still expected to expand, with the majority of the growth expected to continue to come from China.

It is highly unlikely that there will be any greenfield smelter capacity constructed in western countries for the balance of this decade; any new western capacity is expected to be limited to brownfield expansions and debottlenecking.

Over the past decade, in an effort to satisfy growing domestic zinc metal demand, Chinese smelting capacity has increased substantially since 2000.

Currently identified, forecast base case smelter production capability is sufficient to meet forecast demand for refined zinc through to 2019. Thereafter further capacity is required to meet forecast market demand (Figure 19.2).

Figure 19.2 Smelter Capacity



Source: Wood Mackenzie.

Between 2017 and 2021, three zinc smelters in China will enter production adding capacity. In 2014, Rutherford (Mooresboro) smelter in USA started production, replacing the Monaca smelter, however, it was closed in 2015 and it is reported that it will be restarting up in 2019. The Torreon expansion started late last year and is forecast to reach full capacity by the end of 2019.

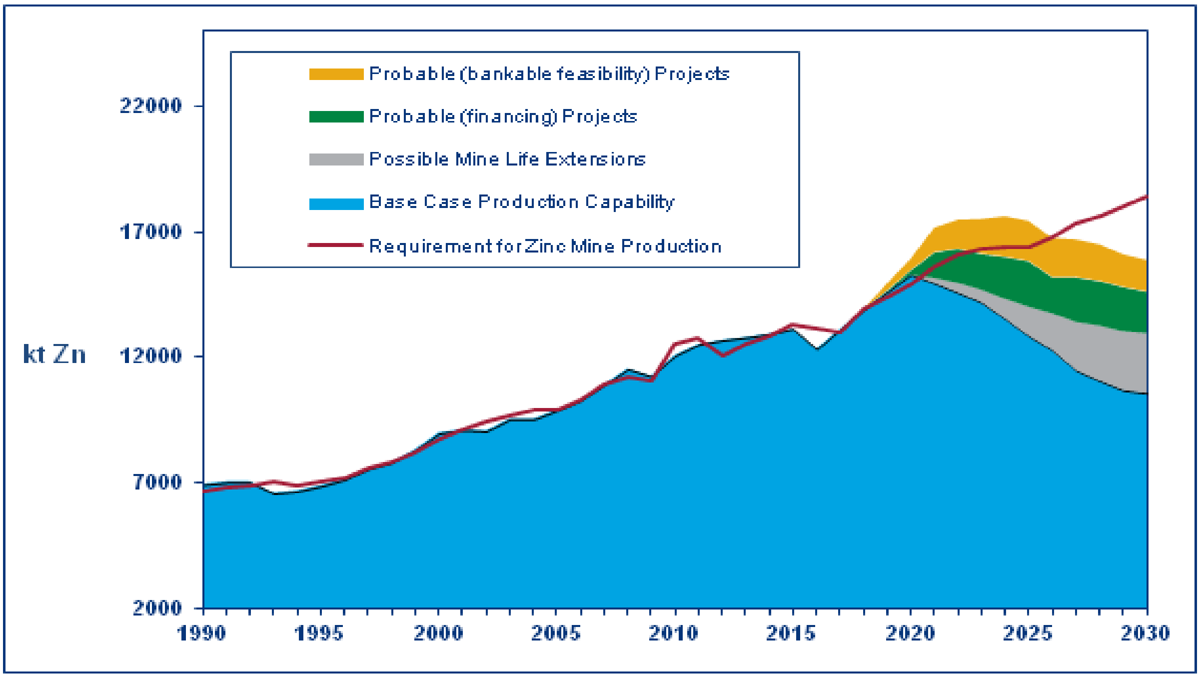
Chinese, and to a lesser extent Indian, smelting companies may continue to expand capacity in an attempt to match growing domestic metal demand. Chinese smelters which are facing increased environmental oversight may not be able to quickly build smelting capacity. It is speculated that while sufficient zinc refining capacity will be available to meet the demand for metal, mine supply may not meet this demand.

19.1.3 Projected Zinc Concentrate Supply/Demand Balance

In 2017, low zinc concentrate stocks constrained the refined production in China. These constraints when combined with a global demand growth of 2.4% are depleting global stocks of refined zinc. Wood Mackenzie have forecasted a fall in global stocks by the start of 2018, and for the period 2017–2022 global growth may grow at an average annual rate of 2.3% p.a, and an average of 1.5% p.a after that. Forecast mine closures and global zinc demand would create an implied shortfall in identified mine output. It is not expected that sufficient new production will be on line before the end of the decade to compensate for the large-scale attrition.

Concentrate shortfalls would translate into significantly reduced metal supply. While improving market fundamentals will support new mine developments, it is not expected by Wood Mackenzie that sufficient production can be brought on stream in the near term to significantly reverse this projected trend. Accordingly, a long-term supply gap is expected to emerge which can only be reversed if prices rise to incentivise development of currently uncommitted projects (Figure 19.3).

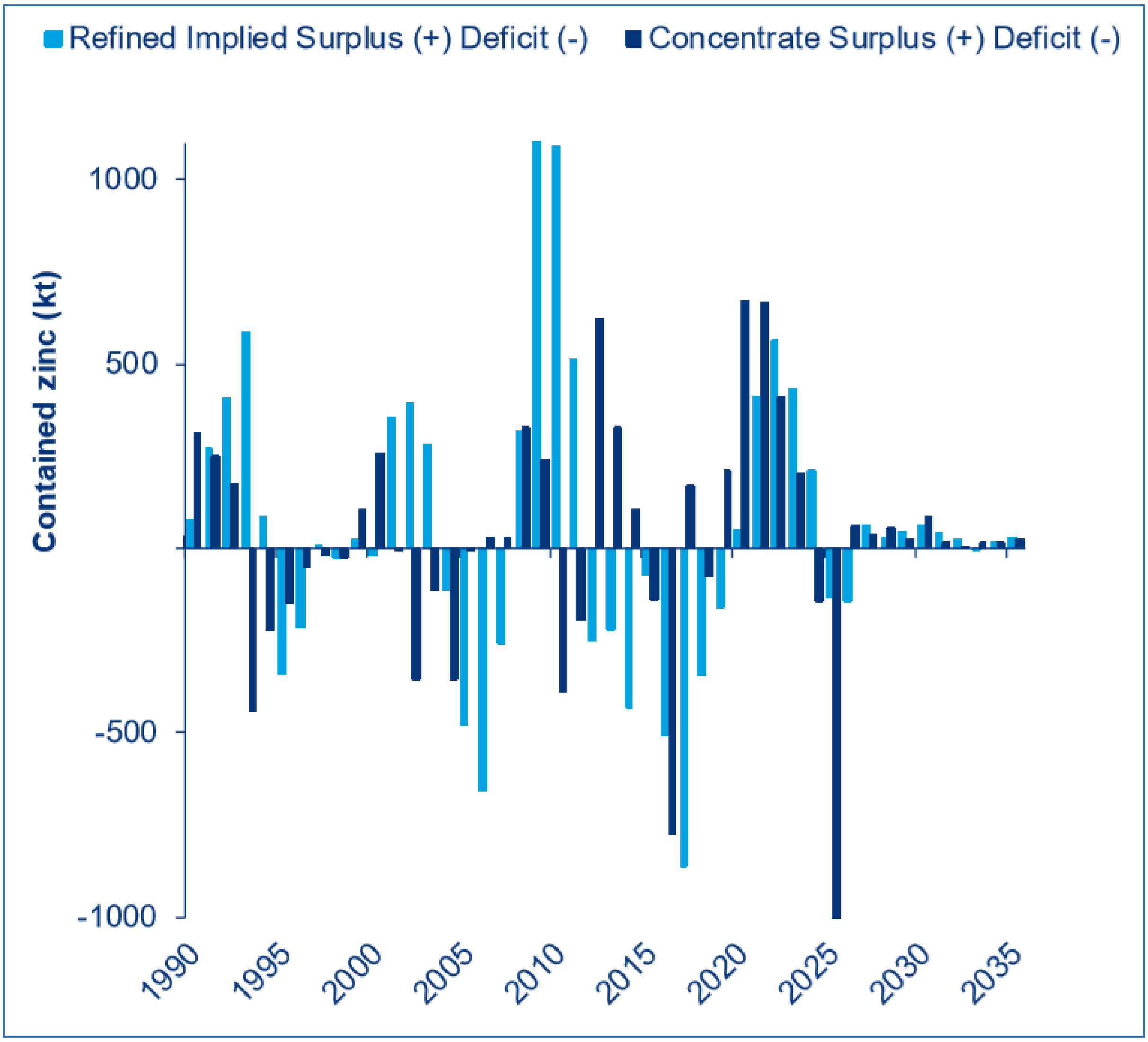
Figure 19.3 Impending Zinc Supply Gap



Source: Wood Mackenzie.

Constraints in the concentrate market limiting metal production, coupled with continued global growth, results in refined zinc stocks being forecast to remain at depressed levels until 2021 (Figure 19.4).

Figure 19.4 Refined Zinc Production Deficit



Source: Wood Mackenzie.

19.1.4 Treatment Charge Outlook

The Kipushi 2017 PFS assumes that zinc concentrate will be sold at industry standard terms. A long-term concentrate treatment charge of $170/dmt concentrate 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. While concentrate grades that fall outside these specifications can often be processed, smelter interest in them may be more-limited because the concentrates will either have to be subject to higher cost processing or blended with other inputs to ensure an appropriate furnace feed mix. Individual smelters may be even more restrictive on certain deleterious elements due to their own particular process technology, feed mix, and/or local regulations.

Penalty rates for impurities in zinc concentrates will vary from smelter to smelter depending on various factors including individual smelter process capabilities, existing capacity for additional inputs of a given impurity and prevailing market conditions.

Precious metal content in concentrates can be a constraining factor as well. While not typically a metallurgical or environmental issue, the presence of high levels of precious metals may be an economic issue for certain smelters / refiners. Not all zinc smelters have precious metal recovery capability (or recoveries may be poor), gold and silver accountabilities in zinc concentrates can vary from buyer to buyer.

Based on the KICO marketing analysis there are no material quality issues foreseen with the concentrates:

- The projected zinc grade will be attractive to smelters.
- The silver and gold levels in the concentrates are projected to be low and below typical smelter payables.
- The projected germanium levels in the concentrate are higher than typical, but are nonetheless, unlikely to be payable as very few zinc smelters actually recover germanium. While germanium may not be 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. Potentially concentrate with low fluorine levels could be purchased and blended to reduce the overall contained fluorine below the penalty threshold.
- Iron and lead levels are both below typical penalty thresholds.

19.2.2 Concentrate Sales Strategy and Distribution

There is currently no African smelter to which the Kipushi concentrates can be reasonably shipped. Although freight differentials will clearly come into play when determining the most suitable buyers for the Kipushi concentrates, the differentials are not deemed wide enough to strongly favour one geographic market over another. Furthermore, with the life-of-mine annual production average of 530 kt concentrate, the Kipushi Project has the potential to be one of the largest zinc mines in the world and should look to have exposure to all the major markets.

Most, if not all traders will offer early payment for concentrates and will typically offer more competitive commercial terms (treatment charges, penalties, etc.) than smelters in exchange for delivery destination options and quotation periods. While the Kipushi concentrates are relatively clean and can likely be placed 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 Project, while taking into consideration prevailing market conditions at the time contract discussions are entered into.

As treatment terms (payable metals, annual treatment charges, escalators, etc.) can be expected to be relatively similar for all buyers of seaborne zinc concentrates, decisions regarding the ultimate distribution of the Kipushi zinc concentrates can focus on desired or preferred partnerships with specific buyers. With treatment terms relatively consistent from one buyer to the next, ocean freight rates should effectively be the only factor significantly differentiating the rates between the alternative destinations.

Although cost differentials are foreseen for deliveries of Kipushi zinc concentrates to the major market destinations, i.e. Europe and Asia, the projected differential is not viewed as significant enough to warrant a focus on one specific geographic region over the other. While consideration should be given to maximising opportunities that may be available in certain markets, (e.g. east coast South America and even North America), for strategic reasons it may be preferable for Kipushi to be active in several different zinc markets.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

This section has not been changed from the Kipushi 2017 Prefeasibility Study and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Kipushi 2017 PFS.

The Kipushi area is of humid subtropical hot summer climate with mild, dry winters and hot humid summers. Rainfall of approximately 1,208 mm is experienced annually in the region of Lubumbashi with the wettest rainfall months occurring from November to April and the driest weather occurs from June to August. The average annual temperatures vary between 14°C and 28°C with average annual relative humidity of 66%.

The Kipushi municipality was originally developed around an existing informally planned village. At the peak of operations, it housed a mine staff of approximately 2,500 workers and their families. The current estimate of the Kipushi population is 150,000 people. As the infrastructure design is based on 20,000 people, there is tremendous pressure on infrastructure, which has not been well maintained.

Kipushi municipality is surrounded by small scale subsistence agriculture, allocated by tribal authorities. Given the population density, there is limited fertile agricultural land available for new allocation. The informal economy in and around Kipushi is driven by small, micro and medium enterprises (SMMEs) who trade in a variety of daily necessities. Artisanal mining of aggregates and retrieving copper from old concentrate run-off also constitute a significant economic activity with an estimated number of 30,000 artisanal miners active in and around the town.

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

Sustainability for the Kipushi Project should focus on the urban population, including continued operation of the potable water pump station, prevention of flooding and water ponding in the community for malaria control and community health initiatives including FIONET. Support and capacity building to SMMEs and to local suppliers to the mine based in Kipushi will be prioritised. There is considerable small-scale agriculture in the impact area, and the possibility of building local capacity to expand to commercial agriculture will be investigated. In addition, support to local schools in the form of bursaries, infrastructure development and collaboration with local Universities will take priority to help develop a young work force with the mine.

20.1 Previous Work

- Environmental Report on the Kipushi Zinc-Copper mine, Democratic Republic of Congo, by The Mineral Corporation, for Kipushi Resources International Limited (KRIL), 2007.
- Etude d'Impact Environnmental et Plan de Gestion Environnmental du Projet (EIA/PGEP), PER 12234, 12349 et 12350 for KICO sprl by DRC Green – EMEC, 2011.
- Environmental Management Plan (EMP) for Tailings Processing Permits PER 12234, 12349 and 12350, by Golder Associates for KICO, 2014.
- Report d'Audit Environnmental in situ Relatif a l'Obtention de l'Attestation de Liberation des Obligations Environnmentales des PER 12234, 12249, et 12250; PE 12434 de la Gécamines Cedes a KICO sprl, Republique Democratiques du Congo, Minitere du Mines, Secretariat General de Mines, Direction de Protection de L'Environnment Miniere, 2011.

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.

In 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. A new tailings storage facility located south of the plant area will be constructed to contain approximately 2 Mt of flotation tailings. All DMS tailings produced from the zinc beneficiation will be used as mine backfill.

20.2 Force Majeure Condition

The legal condition of force majeure on 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 ESHIA Baseline Study

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

- Environmental Baseline (as at November 2011) and Liabilities Assessment.
- Environmental Management Plan (EMPP) Kipushi Tailings, February 2014.
- Assessment of Potable Water Supply infrastructure, August 2012.
- ESHIA Baseline Study, May 2015 including components of:
  - Aquatic Biology Assessment.
  - Visual Baseline.
  - Terrestrial Ecology.
  - Radiological Baseline.
  - Health Impact Assessment.
  - Noise study.
  - Social Risk Assessment.
  - Socio-Economic Baseline.
  - Geochemistry Baseline.
  - Surface Water baseline.
  - Stakeholder Engagement Plan.
  - Groundwater Baseline.
  - Air Quality baseline.
  - Soil and Land-use baseline.

The ESHIA Baseline study used the International Finance Corporation (IFC) guidelines as a standard, which includes the Equator Principles version 3 (EP3); with the exception that no primary health data in the Kipushi impact area were collected.

The primary impacts on the natural and social environment due to mining and related industry were considered to be:

- Air quality: Fugitive dust from historical Tailings Storage Facilities (TSFs), unsurfaced roads, air pollution from vehicle traffic, clay brick firing, veldt fires, and charcoal burning. It was noted in the 2012 report that zinc concentrate was stockpiled on site, with large amounts of mineralised dust present.
- Land use: progressive urbanisation and loss of area available for agriculture, ownership issues, lack of soil fertility (natural), caused (in part) by population influx due to economic opportunities in the mining sector.
- Surface Water: Kipushi mine water discharge is generally within DRC regulatory discharge limits, and there is additional settling and filtering by the wetlands in 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. The CMSK plant has since seized operations and an additional ventilation fan installed.
- Radiation: although localised sources of elevated radiation were identified, the average dose rates fall within the average global dose rates.
- Biological Environment: deforestation and degradation of natural habitat resulting in loss of biodiversity, due to population influx and lack of land management.
- Socio economic environment: economic dependence on mining related business.
- Health Concerns: Malaria remains the highest mortality cause, followed by TB, and STDs (including HIV/AIDS/ARC), exacerbated by poor quality health care, although not a direct impact caused by mining, the loss of the paternal legacy of state owned enterprises increased the concerns.
- Artisanal Miners: volatile and vulnerable group comprising some 20% of the local population as primary or supplementary means of livelihood, KICO has a good working relationship with formalised cooperatives.

20.5 KICO Internal Studies

KICO has also undertaken several studies to complement the Golder ESHIA Baseline Study, including:

- Annual survey of primary, secondary and tertiary schools in the district, including enrolment, available capacity, and tuition fees.
- Socio-economic study of the artisanal mining population.
- SMME survey of local small businesses.
- Survey of health care facilities.
- Survey of Employee's residence locations and proximity to medical service providers.

20.6 Environmental Impact Study Terms of Reference

The terms of reference (ToR) for the update of the environmental impact study (EIS) for the Kipushi Project was compiled by Golder Associates as part of the PFS. The ToR defines EIS update process, provides the project definition, its objectives, the proposed schedule, and identifies potential project impacts in terms of physical, biological, socio-economic and trans-border environments. The ToR is the first step towards obtaining an approved EIA and Environmental Management Plan (EMPP) for the project.

20.6.1 Tailings Management and Disposal

Approximately 2 Mt of flotation tailings will be stored in a new TSF. Several sites provisionally identified for locating the TSF are shown in Figure 20.1. A ranking matrix identified Site 4 as the most optimal TSF location.

The key design features of the TSF are as follows:

- The TSF will be constructed as a full impoundment dam with a compacted earth wall.
- A liner system, including a double layer of 1500 micron HDPE geomembrane, with associated leakage detection, leachate collection system and cushioning layers.
- An elevated toe filter drain, associated toe drain outlets and collection pipeline.
- Stormwater diversion/run-off trenches, to divert rainfall run-off away from the facility.
- Phased construction, with an initial phase of 8.4 m high compacted earth starter impoundment yielding 2.5 years storage capacity. Thereafter the construction of the impoundment walls has been phased, such that the impoundment crest elevation is at least two metres ahead of the tailings, to allow for sufficient freeboard.

Figure 20.1 Potential Tailings Dam Locations – Site 4 Selected for the Study

20.7 Water Management

20.7.1 Regional Surface Water Resources

The mine is located in the upper reaches of the Kipushi catchment with the existing mine tailing storage facilities located in the middle reaches of the Kipushi River. The Kanyameshi River joins the Kipushi River from the north about 3 km downstream of the TSF. The Kipushi River flows east for another 1 km before it joins the Kafubu River. The Kamalenge River flows in an easterly direction to the north of the Kipushi River catchment. The Kamalenge River is also a tributary of the Kafubu River. The Kamalenge Lake is located in the upper reaches of the Kamalenge River (also referred to as Lac Kipushi). A small area of the mine is located in the Kamalenge River catchment with the run-off draining to the Kamalenge Lake. The Kafubu River drains in a southerly direction and turns to flow in an easterly direction at the confluence of the Kafubu and Kipushi Rivers. There are extensive wetlands in the lower reaches of the Kipushi River and in the Kafubu River downstream of the Kafubu and Kipushi River confluence. The Kafubu River flows north-east towards Lubumbashi. Water is abstracted from the river to supply Lubumbashi and is used for irrigation. Figure 20.2 details the catchment areas of the rivers.

Figure 20.2 Location and Extent of the Surface Water Catchments in the Vicinity of Kipushi Mine

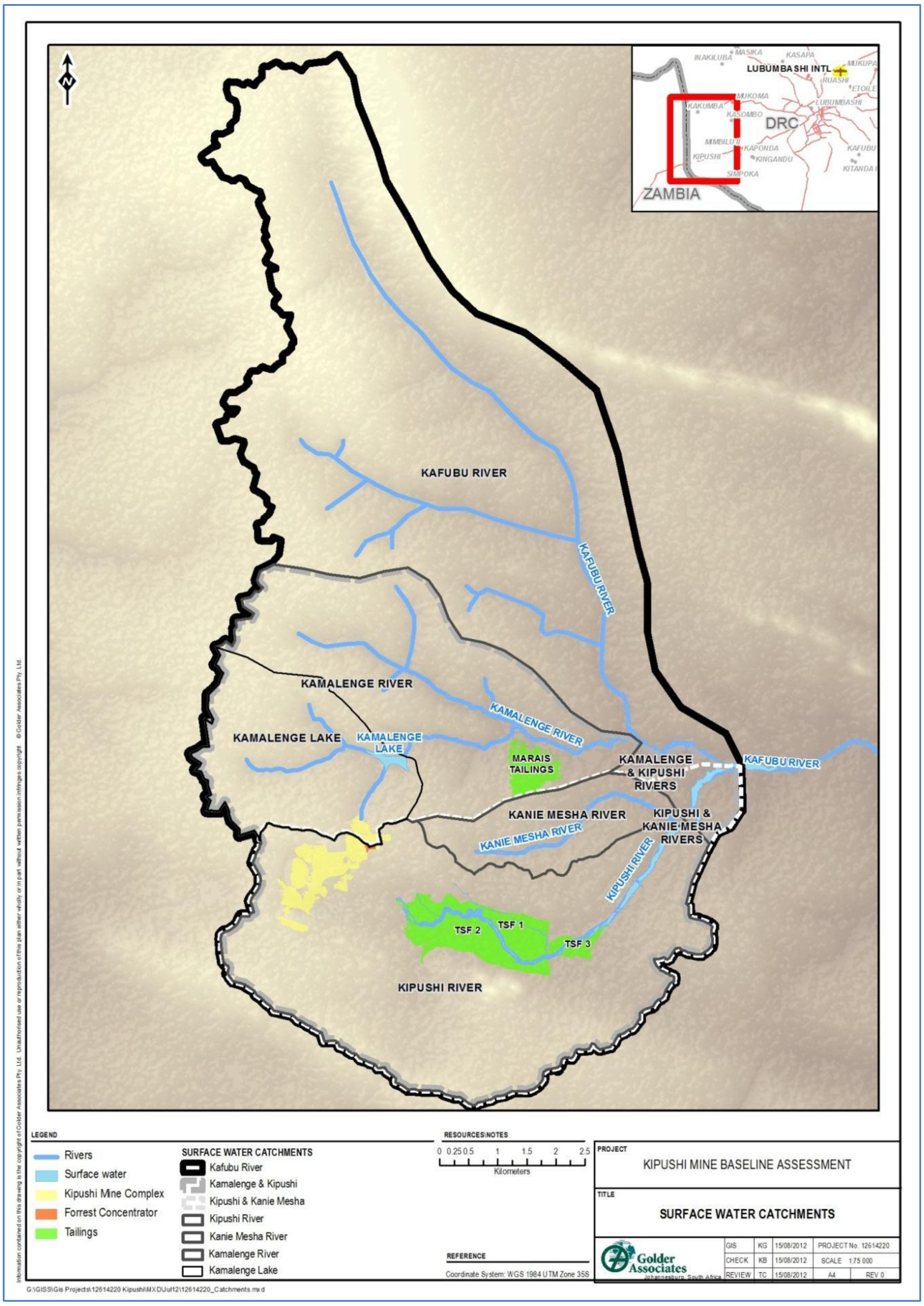


Figure by Golders, 2012.

20.7.2 Mine Stormwater Management

The layout of the mine stormwater management drains is shown in Figure 20.3. Historically the stormwater run-off, and water pumped from underground, is conveyed in channels to discharge into the Kipushi River to the east of the mine complex. The drain from Shaft 5 was used to convey tailings from the CMSK concentrator for deposition on the TSF. However, the CMSK has now seized operations. The proposed development consists of a new TSF, plant, stockpile and waste rock storage facility. Stormwater run-off from the new infrastructure will report to the existing stormwater drainage system.

There are four main stormwater drainage channels on surface. Drain locations are shown in Figure 20.3.

Figure 20.3 General Layout of Mine Infrastructure and Candidate TSF Site

20.7.3 PFS Mine Water Circuit

Underground water is planned for use as process water in the new process plant. Flotation tailings will be deposited in a new tailings storage facility (TSF), located south of the process plant.

In the proposed scheme (Figure 20.4), the return water from the TSF is first neutralised with lime ($Ca(OH)_2$) and blended with the excess underground water before being discharged into the Kipushi river, via the north cut-off channel.

A neutralisation plant has been included in the PFS, on the basis that the plant metallurgical simulations undertaken, suggest that the pyrite to dolomite content of the tails is such, that the TSF return water is likely to be acidic, and that even after blending with underground water prior to discharge, the water released to the environment would fall outside the DRC prescribed pH discharge limits.

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

Water supply for the Kipushi area is obtained from a well field located approximately 1.0 km south-east of the town and south of the tailings dam. The well field was designed to have 10 large diameter boreholes drilled into the Kakontwe Dolomite/ Limestone aquifer. Six of these boreholes were equipped with pumping equipment and the other four were left unequipped to be standby wells. The pumps installed are of the vertical spindle type, where the pump is at the bottom of the borehole and is driven by a shaft connected to an electrical motor on surface. Water is delivered to a Central Sump.

Potable water is received from the local municipal supply, stored in the new potable water tank and distributed to various users.

Figure 20.4 Water Management Block Flow Diagram

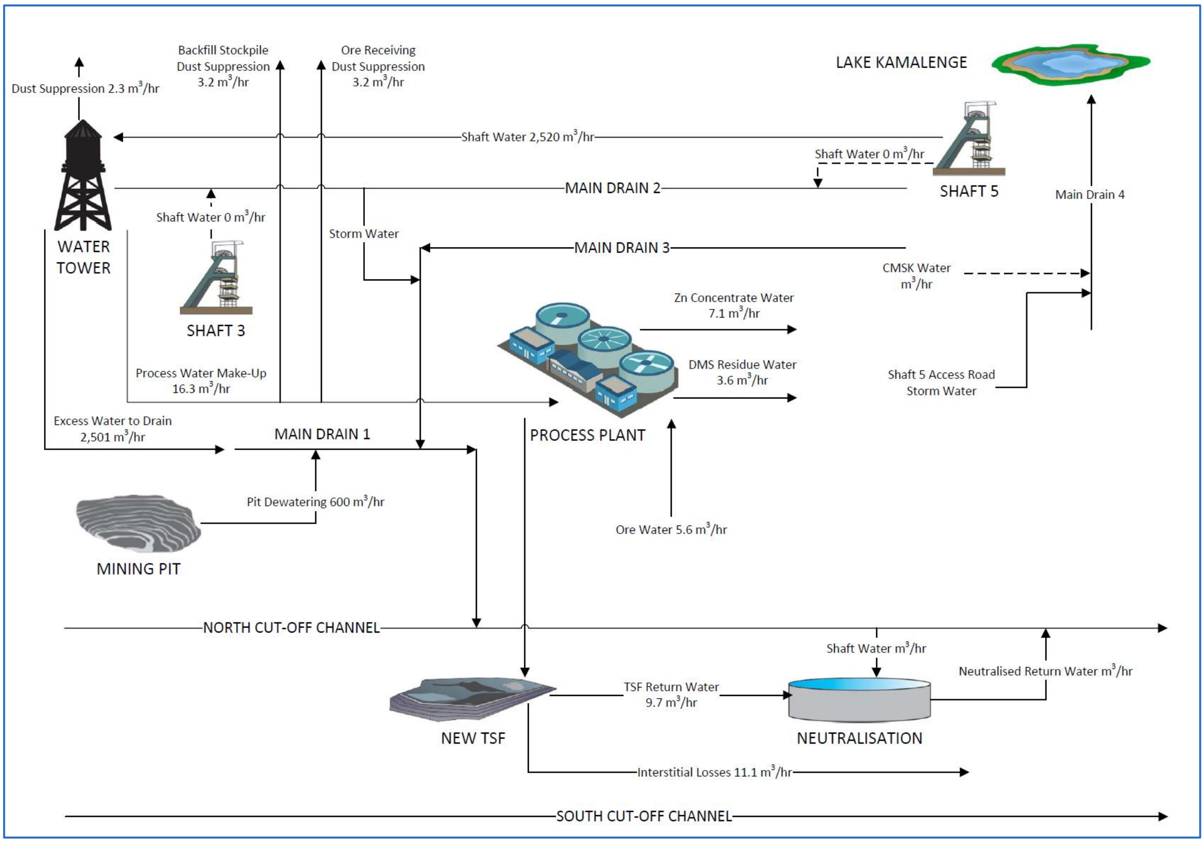


Figure by KICO, 2018.

20.8 Mine Closure Analysis

A closure scenario was developed for Kipushi Mine using a snapshot of three different time periods as explained below and reflected in Table 20.1.

- A snapshot view of the site on the last day of operations, assuming full life-of-mine and the context in which decommissioning and closure activities will follow.
- Key activities/actions during the decommissioning and closure period.
- The anticipated post closure character/nature of the rehabilitated site, and remaining activities to be implemented to progress the site to a stable and self-sustaining state, for eventual site relinquishment.

Table 20.1 Kipushi Project Closure Scenario

On Last Day of Operations	During Decommissioning and Closure	Post Closure
• Mining would have ended and Shaft P2 and Shaft 5 will become available for rehabilitation. Kipushi will have limited/no stockpiles left and the plant will have been run down and be available for demolition/dismantling. • The TSF will be at full capacity and tailings deposition will have ended. • Product export by rail would have ceased and the railway siding will become defunct. • Responsibilities for rehabilitation would have been clearly defined in terms of agreements already in place. • The use of dedicated waste cells constructed within the TSF would have been implemented during operations for the disposal of demolition and other waste as necessary.	• Demolition of all infrastructure not earmarked for reuse will take place and the resulting footprint areas will be rehabilitated. Infrastructure to be demolished and rehabilitated broadly includes the plant, all on-site buildings, stockpiles, conveyors, rail siding, Shaft 2 and Shaft 5 and related infrastructure. • Substations, transformers, switchyards, powerlines and roads will be handed over to government for management. • Demolition waste will be decontaminated within the dedicated decontamination bay/area and disposed within the onsite waste cell. Benign concrete waste will also be used for infilling of cavities created by infrastructure demolition. • Any contaminated soils found within the plant area will be excavated and disposed of within the TSF. This could be an additional cell. • Decontaminated steel and related material from plant demolition, having salvage value will remain on-site for sale. • Any hazardous waste (if any) will be transported by road to South Africa and be disposed of at a registered hazardous landfill site. • Site drainage lines will be reinstated on the rehabilitated surface areas to ensure the site is free draining and to limit/avoid ponding. • The outlet and penstock of the TSF will be plugged and sealed. After the disposal of demolition waste and contaminated soil the upper surface of the cells will be aligned to the slope of the operationally created upper surface beach of the TSF. A borrow pit will be established to obtain waste rock for the upper surface cross walls construction. The concentric cross walls will be constructed using rock grid as a support layer on the upper surface of the tailings at a spacing of approximately 30m. Topsoil will be placed on the outer slopes of the TSF and revegetated. An emergency spillway from the upper surface to ground level will also be constructed.	• Monitoring will take place to confirm success of closure measures implemented at the site, until performance objectives and abandonment criteria are met. Surface water, groundwater and rehabilitation monitoring to be conducted. • Care and maintenance will be implemented and further guided based on monitoring results. • Site relinquishment could be considered based on demonstration of success of the rehabilitation effort.

20.9 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.
- Bursaries for high performance mathematics and science students in local high schools in Kipushi.
- Student apprenticeships from technical schools in Kipushi, for training in the machine, garage and welding shops.
- Support to the FIONET malaria diagnostics system implementation, to be installed at 42 health care facilities in the impact Kipushi Health Zone.
- Collaboration with the Municipal authorities on road maintenance, and infrastructure support for municipal buildings.
- Ad hoc school repair programmes.

20.10 Environmental and Social Studies Going Forward

An updated Environmental and Social Impact Assessment is planned as part of the feasibility study. The results of this study will inform the ongoing Environmental Management Plan and provide a starting point for the Sustainable Social Development Plan for the life-of-mine.

21 CAPITAL AND OPERATING COSTS

This section has not been changed from the Kipushi 2017 Prefeasibility Study and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Kipushi 2017 PFS.

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 20%. Additional detail work is required to define the costs. All monetary figures expressed in this report are in US dollars (US$) unless otherwise stated. The cost estimates have an overall accuracy provision 25%. Costs have a base date of Q1 2017.

21.1 Capital Cost

The total Project direct capital cost estimates are shown in Table 21.1. Capital costs have been estimated separately for each area based on the quantities and design criteria.

Table 21.1 Total Project Capital Cost

Item	Pre-Production ($M)	Production ($M)	Total ($M)
Mining			
Underground Mine Refurbishment	17	–	17
Underground Mining	57	128	185
Capitalised Mining Operating Costs	37	–	37
Subtotal	**112**	**128**	**239**
Process and Infrastructure			
Process and Infrastructure	78	7	84
Rail	32	–	32
Capitalised Processing	7	–	7
Subtotal	**116**	**7**	**123**
Closure			
Closure	–	20	20
Subtotal	**–**	**20**	**20**
Indirects			
EPCM	12	–	12
Capitalised G&A	11	–	11
Subtotal	**23**	**–**	**23**
Others			
Owners Cost	11	–	11
Studies	5	–	5
Kico 2018 Site	33	–	33
Sustaining	–	24	24
Capital Cost Before Contingency	**300**	**178**	**478**
Contingency	37	–	37
Capital Cost After Contingency	**337**	**178**	**515**

Note: 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 mining costs were applied to the financial model as operating costs or capital costs. In the mining cost model, costs are broken down into specific areas including development, load and haul and production. The KICO 2018 budget has been accounted for with the portion in addition to calculated capital included as KICO 2018 Site. The operating cost summary can be seen in Figure 21.1. The capital cost summary can be seen in Figure 21.2.

The contractor is responsible for development and production costs. Including but not limited to the decline, level development, stopping, and backfilling. The crusher, pumps and winders will be operated by employees that are directly employed by KICO. The mining equipment will be owned by KICO however the contractor will be responsible for operation and maintenance.

The estimating methodology applied in the development of the cost estimates, is in line with industry accepted norms for a PFS / Class IV estimate. The estimated capital cost for the process plant and surface infrastructure accounts for:

- New conveyor connecting Shaft 5 to the process plant ROM.
- ROM stockpiling.
- New process plant and associated in-plant infrastructure, including laboratories.
- General infrastructure such as electrical substations, MCCs, fuel systems, office buildings, workshops, roads, overhead lines etc.
- Earthworks and terracing.
- Tailings storage facility.
- Rail loading terminal.

The estimated capital cost was derived from budget quotations received from various equipment suppliers, package pricing for specific areas of the plant and in-house database pricing for minor equipment items.

Supply, install and preliminary and general (P&G) costs by area and by discipline, were factored off the area mechanical equipment supply costs. Earthworks and civils costs were based on preliminary geotechnical work, preliminary bills of quantities (BOQ) and supply and install rates supplied by contractors based in Lubumbashi.

The pricing for new buildings was based on budget quotations, whilst the costs for refurbishing buildings was based on preliminary BOQ's, building supply and refurbishment rates supplied by contractors local to Lubumbashi and derived from recent projects in the DRC.

21.2 Operating Costs

Operating costs have been estimated from labour numbers and current labour rates, equipment operating costs, consumable and other materials costs, power, fuel and other estimates. The operating cost estimates have been presented in Table 21.2.

Table 21.2 Estimated Operating Costs

Description	Total ($M)	5-Year Average	LOM Average
		($/t Milled)	
Site Operating Costs:			
Mining	415	52	48
Processing Zn	194	23	23
General and Administration	144	17	17
Total	**753**	**93**	**88**

21.3 Mining Cost Summary

The mining costs were applied to the financial model as operating costs or capital costs. In the mining cost model, costs are broken down into specific areas including development, load and haul and production. The KICO 2018 budget has been accounted for with the portion in addition to calculated capital included as KICO 2018 Site. The operating cost summary can be seen in Figure 21.1. The capital cost summary can be seen in Figure 21.2.

The contractor is responsible for development and production costs. Including but not limited to the decline, level development, stopping, and backfilling. The crusher, pumps and winders will be operated by employees that are directly employed by KICO. The mining equipment will be owned by KICO however the contractor will be responsible for operation and maintenance.

Mining operating costs include:

- Development.
- Production.
- Load and haul.
- Labour.
- Main pumping system.
- Big Zinc stope pumping.
- Other indirects.
- Backfill.

Figure 21.1 Mining Operating Cost Summary

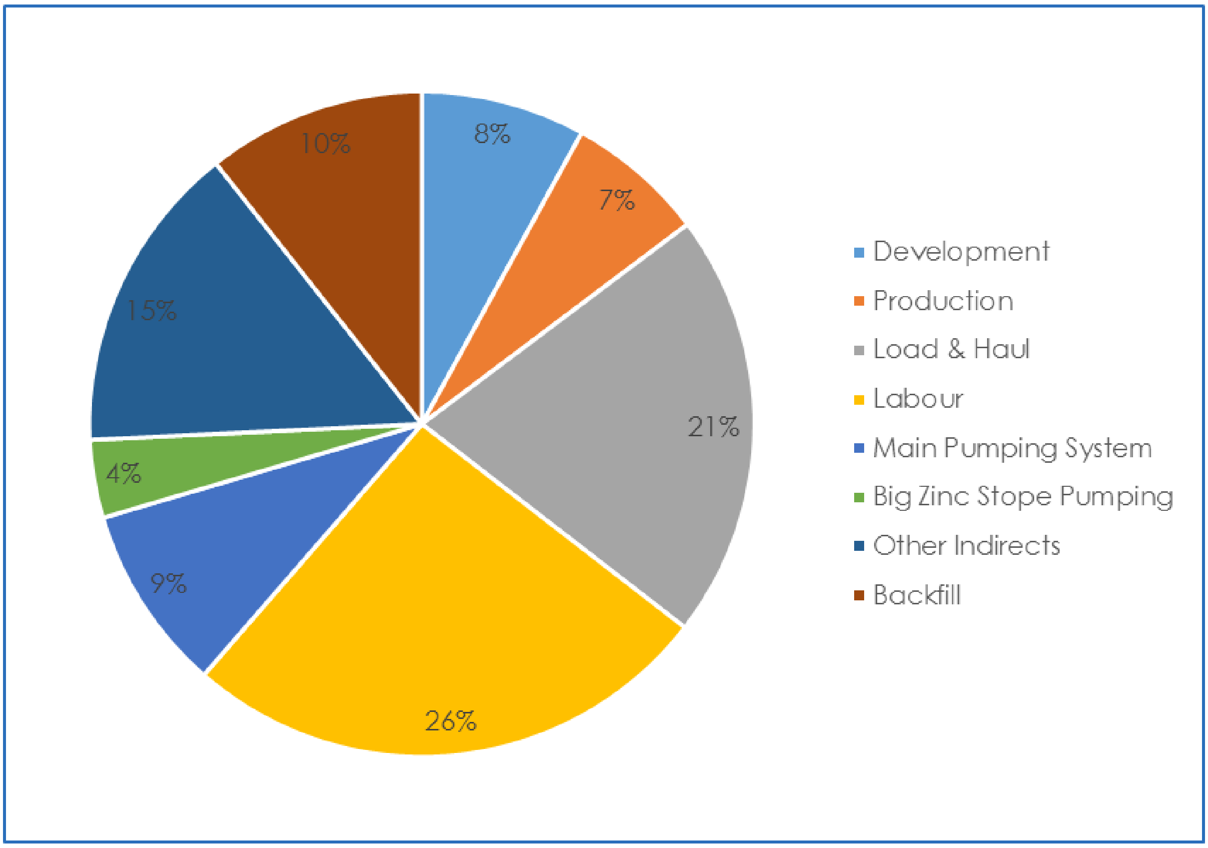


Figure by OreWin, 2017.

Mining capital costs include:

- Development.
- Load and haul.
- Labour.
- Underground fixed equipment.
- Underground mobile equipment.
- Office and supply.
- KICO 2018 site.
- Mine rehabilitation.
- Studies.

Figure 21.2 Mining Capital Cost Summary

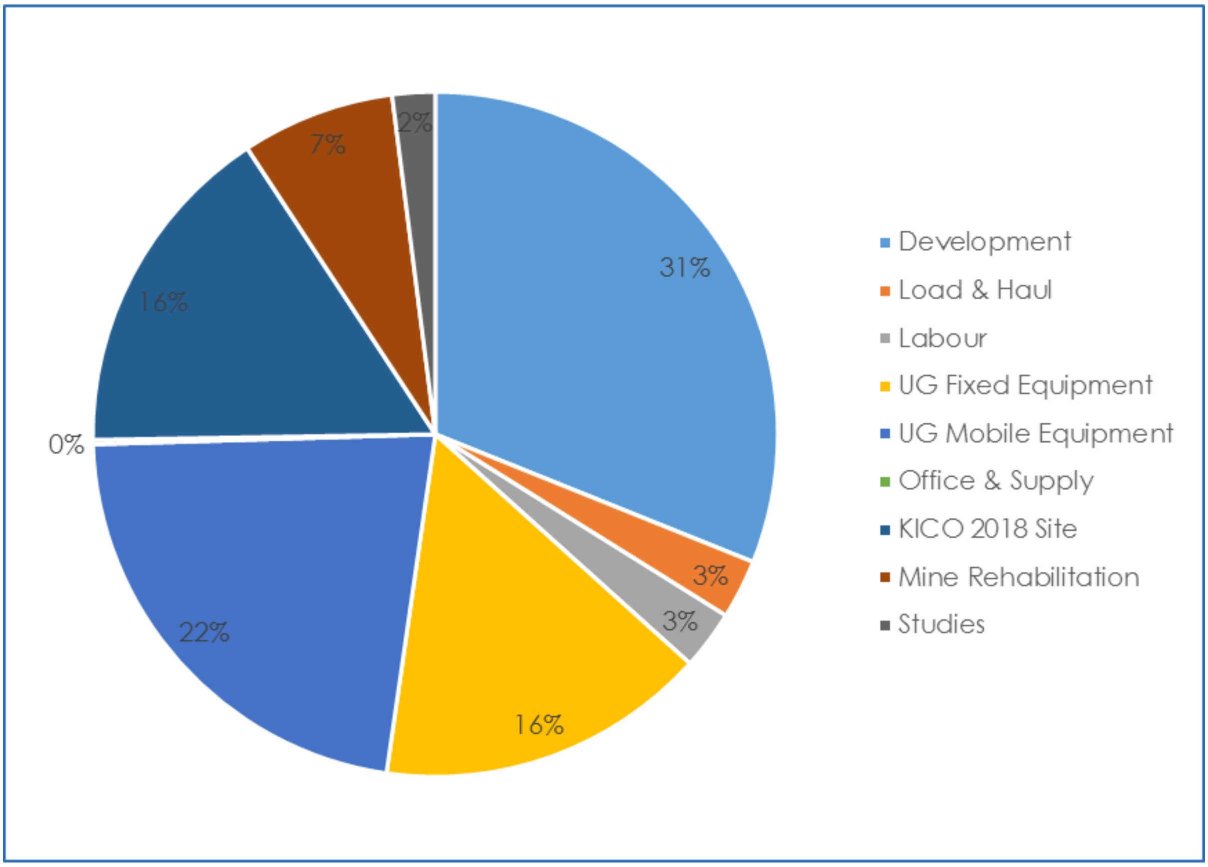


Figure by OreWin, 2017.

21.4 Process and Infrastructure Cost Summary

The process and infrastructure were prepared by MDM. The estimating methodology applied is in line with industry accepted norms for a PFS estimate. The following has been included in the capital costs for process plant cost estimates:

- Ore receiving and crushing.
- DMS.
- Milling.
- Flotation.
- Concentrate, thickening, filtration and packaging.
- Waste management.
- TSF.
- Utilities and services.
- Reagents.
- Plant infrastructure.
- Plant mobile equipment.
- Spares, first fills and bonds (equipment, reagents and consumables first fills, commissioning spares).

The following has been included in the capital costs for infrastructure cost estimates:

- Bulk services.
- Site preparation.
- Buildings and structures (new and refurbished).
- Communications.
- IT hardware and software.
- Security and access control.
- Site Costs.
- Mobile equipment.
- Services contracts.
- Community Support.

The following has been included in the operating costs for infrastructure cost estimates:

- Plant consumables.
- Crusher Consumables.
- Screens.
- DMS Cyclones.
- Mill Balls - Grinding Media.
- Filters.
- Packaging Plant Bags.
- Plant reagents: FeSi, flocculant, flotation reagents.
- Plant mobile equipment.
- Plant maintenance.
- Power.
- Labour.
- Production and dispatch.
- Plant and infrastructure day work services.
- Shift maintenance.
- Laboratory service level agreement.
- TSF water treatment.

The breakdown of the process operating costs can be seen in Figure 21.3.

Figure 21.3 Process Operating Cost Summary

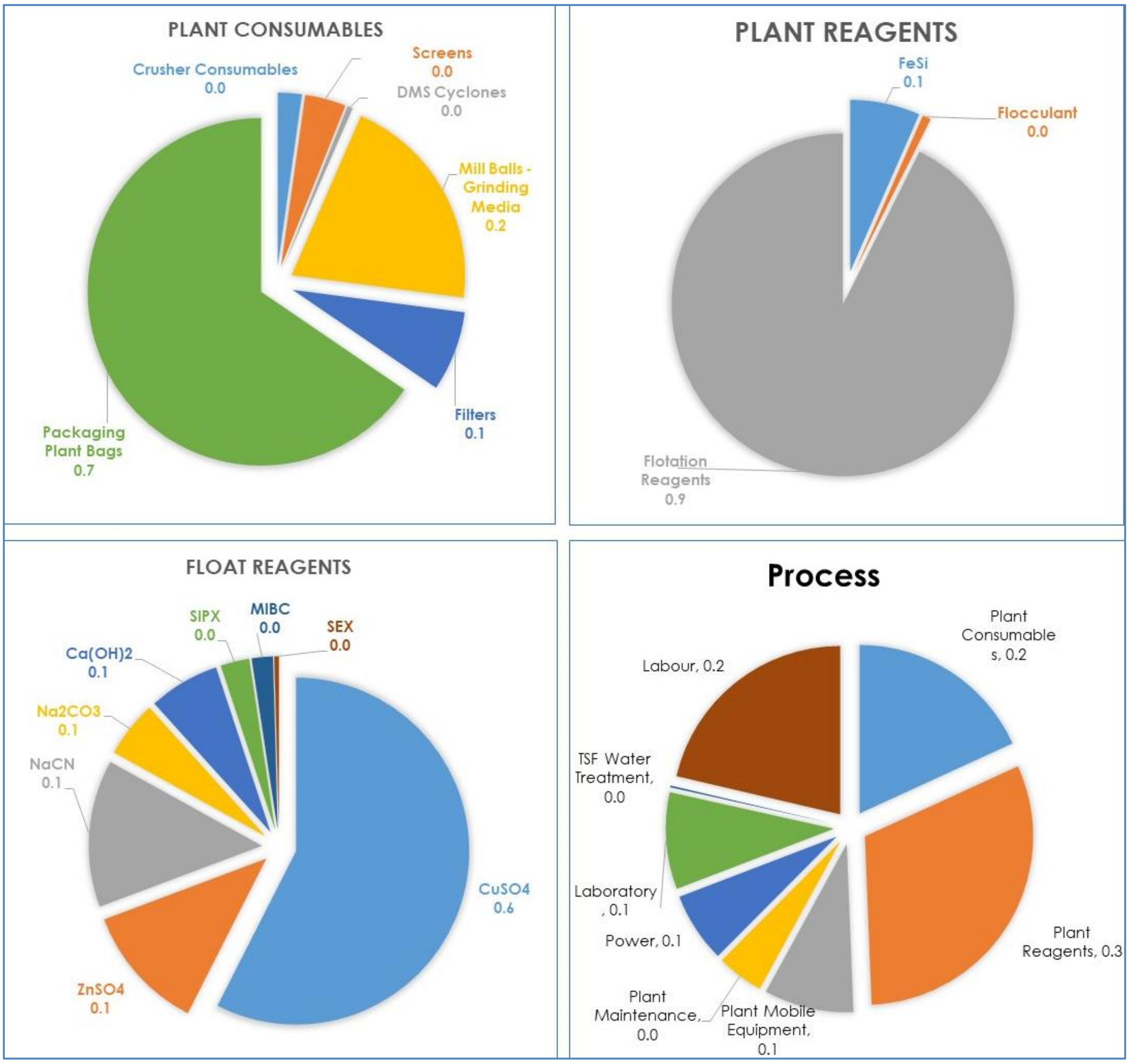


Figure by MDM, 2017.

21.5 General and Administration Cost Summary

The General and Administrative (G&A) costs include costs not directly attributable to operational output such as the mining and processing operations, as shown in Figure 21.4.

The following costs have been included in total G&A cost:

- Office and general expenses.
- Maintenance and inspection contracts.
- Equipment and sundry.
- Fuels and utilities.
- Rentals and leases.
- Insurance and insurance taxes.
- IT hardware and software.
- Personnel transport.
- Communications.
- Licenses and land fees.
- Labour.
- Accommodation and messing.
- Medical support.
- Expatriate flights.
- Light vehicles.
- Environmental, community development and engagement.
- Banking and audit fees.
- Legal and consultants.

Figure 21.4 General and Administration Cost

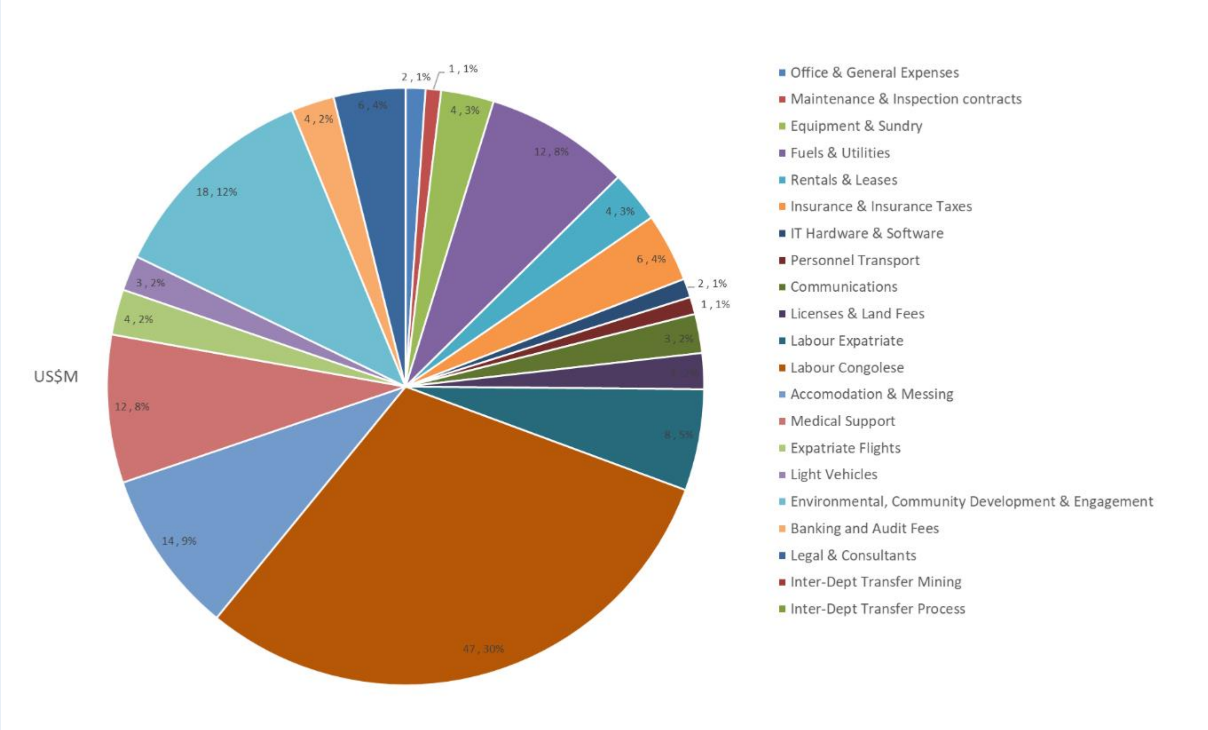


Figure by OreWin, 2017.

21.6 Owners Cost Summary

The owner's costs are 10% of the plant and infrastructure, rail infrastructure, and tailings storage facility capital costs.

21.6.1 Concentrate Transport Costs

The costs for transport from Kipushi via Durban in South Africa to China (including all taxes) is estimated to total $212.25/t wet 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.

21.6.2 Rail Refurbishment Costs

Repair and refurbishment costs for the approximate 34 km of track between Kipushi and Munama were prepared by Grindrod. The estimated cost to rebuild the Kipushi to Munama line including the partial rebuilding of the Kipushi Station, shown in Table 21.3.

Table 21.3 Cost Estimate Kipushi to Munama

Item	Cost Estimate US$M
Construction including Kipushi station	$23.5
Design and supervision @10%	$2.4
Subtotal	$25.9
Contingency @15%	$3.8
Total	**$29.7**

22 ECONOMIC ANALYSIS

This section has not been changed from the Kipushi 2017 Prefeasibility Study and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Kipushi 2017 PFS.

22.1 Production and Cost Summary

All monetary figures expressed in this report are in US dollars (US$) unless otherwise stated. The Kipushi Project financial model is presented in 2017 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 2017 PFS are summarised in Table 22.1. The mining production forecasts are shown in Table 22.2 and forecast zinc tonnes mined are shown in Figure 22.1. The processing tonnes and concentrate and metal production are summarised in Figure 22.2 and Figure 22.3 respectively.

Table 22.1 Kipushi 2017 PFS Results Summary

Description	Unit	Total
Zinc Feed - Tonnes Processed		
Quantity Zinc Tonnes Treated	kt	8,581
Zinc Feed grade	%	32.14
Zinc Recovery	%	89.61
Zinc Concentrate Produced	kt (dry)	4,196
Zinc Concentrate grade	%	58.91
Metal Produced		
Zinc	Mlb	5,449
Key Cost Results		
Pre-Production Capital	US$M	337
Mine Site Cash Cost	US$/lb Payable Zn	0.14
Realisation	US$/lb Payable Zn	0.35
Total Cash Costs	US$/lb Payable Zn	0.48
Site Operating Costs	US$/t milled	87.77

Table 22.2 Mining Production Statistics

	Unit	Total LOM	5-Year Average	LOM Average
Zinc Feed - Tonnes Processed				
Quantity Zinc Tonnes Treated	kt	8,581	777	780
Zinc Feed grade	%	32.14	30.20	32.14
Zinc Recovery	%	89.61	88.76	89.61
Zinc Concentrate Produced	kt (dry)	4,196	354	381
Zinc Concentrate grade	%	58.91	58.51	58.91
Metal Produced				
Zinc	kt	2,472	207	225

Figure 22.1 Zinc Tonnes Mined

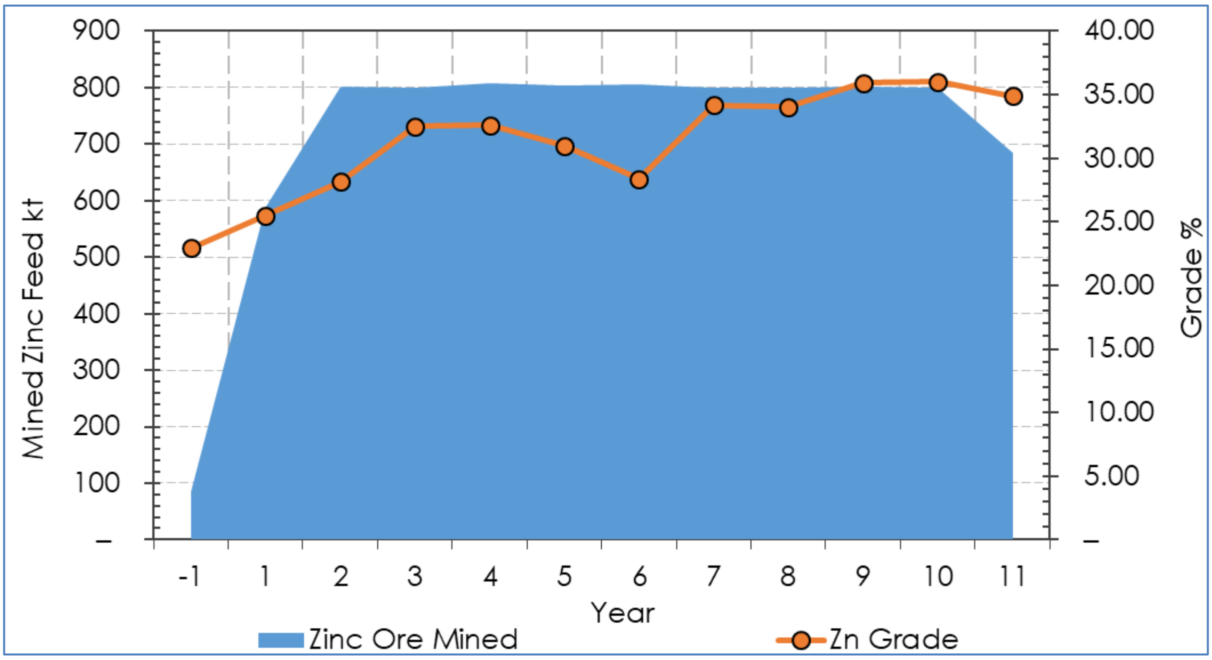


Figure by OreWin, 2017.

Figure 22.2 Zinc Tonnes Processed

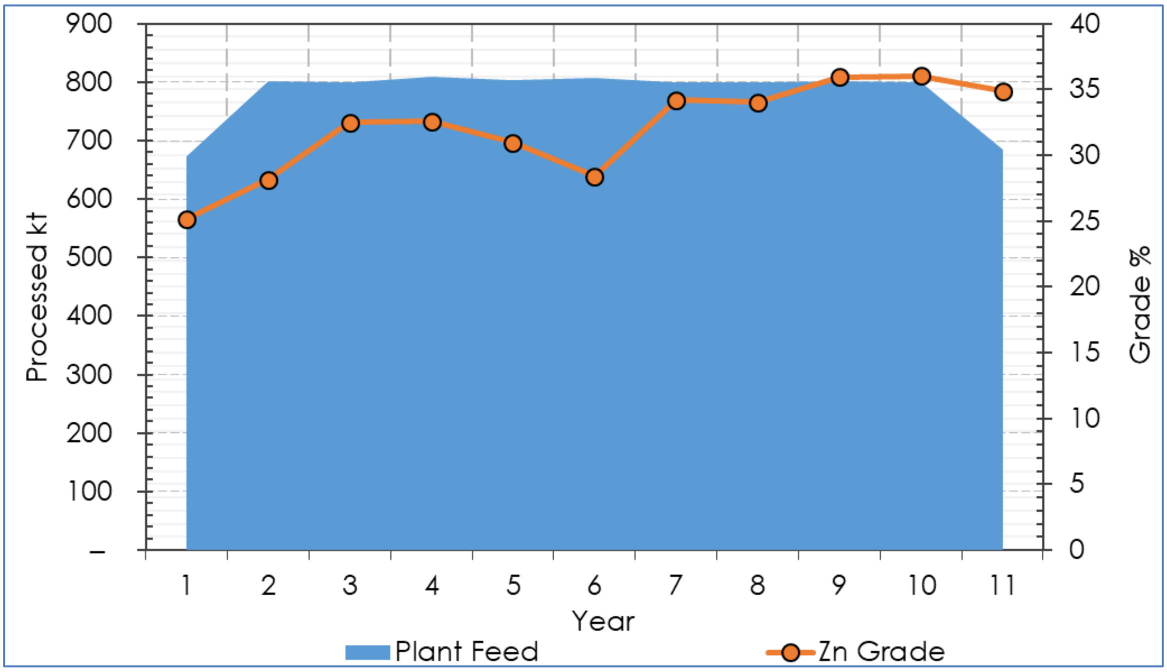


Figure by OreWin, 2017.

Figure 22.3 Concentrate and Metal Production

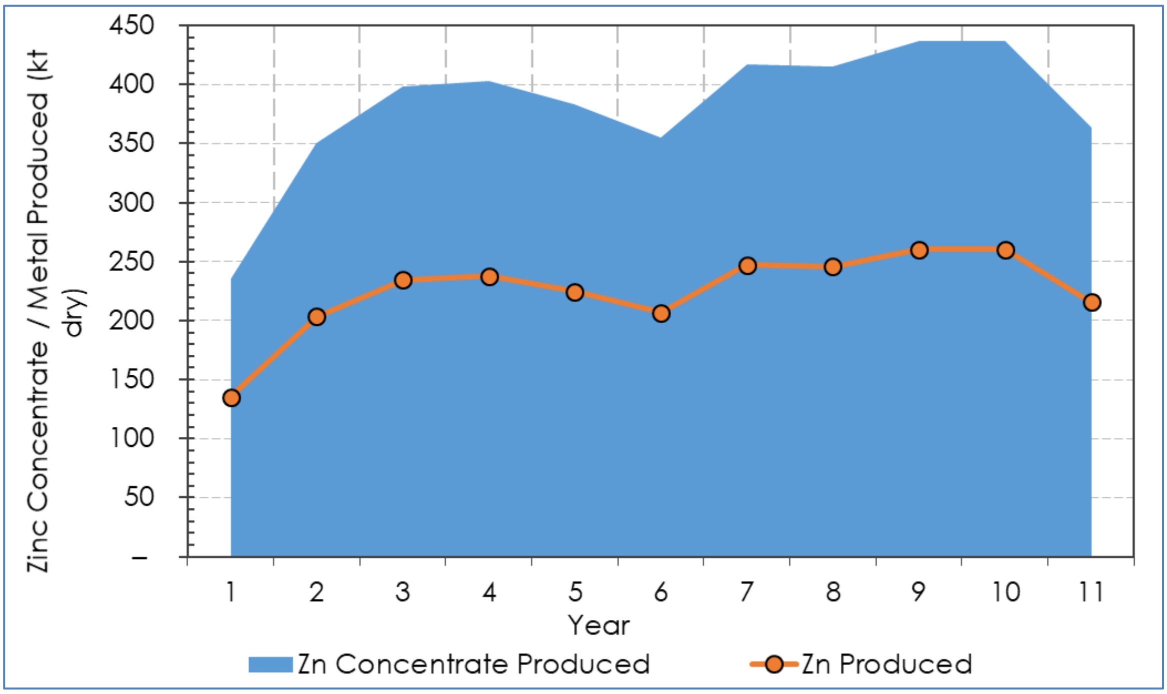


Figure by OreWin, 2017.

22.2 Project Financial Analysis

The estimated Mine site cash costs are shown in Table 22.3. Total estimated cash costs for the first five years of production are $1,105/t zinc and the average for the life of the mine is $1,066/t zinc. Zinc provides the only revenue included in the analysis. There are no credits from other metals included in the cash cost. These estimated costs include only direct operating costs of the mine site, namely:

- Mining.
- Concentration.
- Tailings.
- General and administrative (G&A) costs.
- Government fees and charges (excluding corporate taxation).

The projected financial results include:

- After-tax net present value (NPV) at an 8% real discount rate is $683M.
- After-tax internal rate of return (IRR) is 35.3%.
- After-tax project payback period is 2.24 years.

Table 22.3 Cash Costs

Description	5-Year Average	LOM Average
	($/lb Zn)	
Mine Site Cash Cost	0.16	0.14
Realisation	0.34	0.35
Total Cash Costs Before Credits	**0.50**	**0.48**

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 $2,425/t Zn. The prices are based on a review of consensus price forecasts from a financial institutions and similar studies that have recently been published. The estimated total Project direct capital costs are shown in Table 22.5.

Table 22.4 Operating Costs and Revenues

Description	Total ($M)	5-Year Average ($/t Milled)	LOM Average ($/t Milled)
Revenue:			
Gross Sales Revenue	**5,095**	**550**	**594**
Less Realisation Costs			
Transport Costs	972	103	113
Treatment and Refining Charges	713	77	83
Royalties	197	21	23
Total Realisation Costs	**1,883**	**202**	**219**
Net Sales Revenue	**3,212**	**348**	**374**
Less Site Operating Costs			
Mining	415	52	48
Processing Zn a	194	23	23
General and Administration	144	17	17
Total	**753**	**93**	**88**
Operating Margin ($M)	**2,459**	**255**	**287**
Operating Margin (%)	**48.2**	**46.4**	**48.2**

Table 22.5 Total Project Capital Cost

Item	Pre-Production ($M)	Production ($M)	Total ($M)
Mining			
Underground Mine Refurbishment	17	–	17
Underground Mining	57	128	185
Capitalised Mining Operating Costs	37	–	37
Subtotal	**112**	**128**	**239**
Process and Infrastructure			
Process and Infrastructure	78	7	84
Rail	32	–	32
Capitalised Processing	7	–	7
Subtotal	**116**	**7**	**123**
Closure			
Closure	–	20	20
Subtotal	**–**	**20**	**20**
Indirects			
EPCM	12	–	12
Capitalised G&A	11	–	11
Subtotal	**23**	**–**	**23**
Others			
Owners Cost	11	–	11
Studies	5	–	5
Kico 2018 Site	33	–	33
Sustaining	–	24	24
Capital Cost Before Contingency	**300**	**178**	**478**
Contingency	37	–	37
Capital Cost After Contingency	**337**	**178**	**515**

Note: 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 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 22.10.

Table 22.6 Financial Results

	Discount Rate	Before Taxation	After Taxation
Net Present Value ($M)	Undiscounted	1,944	1,435
	5.0%	1,239	900
	8.0%	953	683
	10.0%	743	517
	12.0%	628	431
	15.0%	487	325
	18.0%	401	262
	20.0%	335	213
Internal Rate of Return	–	41.7%	35.3%
Project Payback Period (Years)	–	1.9	2.2

Table 22.7 Economic Assumptions

Parameter	Unit	Financial Analysis Assumption
Zinc Price	US$/lb	1.10
Zinc Treatment Charge	$/t concentrate	170.00

Table 22.8 After Tax NPV_8 Sensitivity to Zinc Price and Discount Rates

Discount Rate (%)	Zinc (US$/lb)								
	0.80	0.90	1.00	1.10	1.20	1.40	1.50	1.70	2.00
Undiscounted	516	823	1,129	1,435	1,742	2,355	2,661	3,274	4,193
5%	254	472	687	900	1,111	1,533	1,744	2,165	2,796
8%	150	331	508	683	855	1,199	1,370	1,713	2,226
10%	96	257	414	568	719	1,021	1,172	1,473	1,923
12%	51	195	335	471	605	872	1,005	1,271	1,668
15%	-2	121	239	354	467	691	802	1,025	1,357
18%	-42	63	164	262	358	548	642	831	1,112
20%	-64	32	124	213	299	470	555	724	977

Note: Table shows NPV_8 $M.

Table 22.9 After Tax NPV_8 and IRR Sensitivity to Zinc Price and Zinc Treatment Charge

Zinc Treatment Charge (US$/t)	Zinc Price (US$/lb)								
	0.80	**0.90**	**1.00**	**1.10**	**1.20**	**1.40**	**1.50**	**1.70**	**2.00**
50.00	347	524	698	870	1,043	1,385	1,557	1,899	2,412
	23.1%	29.8%	35.8%	41.3%	46.5%	56.0%	60.5%	69.0%	80.5%
100.00	266	444	619	792	965	1,308	1,479	1,822	2,334
	19.8%	26.9%	33.2%	38.8%	44.2%	53.9%	58.4%	67.2%	78.8%
150.00	183	364	540	714	886	1,230	1,401	1,744	2,257
	16.3%	23.8%	30.4%	36.3%	41.8%	51.7%	56.4%	65.2%	77.1%
170.00	150	331	508	683	855	1,199	1,370	1,713	2,226
	14.9%	22.5%	29.2%	35.3%	40.8%	50.9%	55.5%	64.4%	76.4%
200.00	99	282	461	635	808	1,152	1,324	1,666	2,179
	12.6%	20.5%	27.4%	33.7%	39.3%	49.6%	54.3%	63.2%	75.4%
250.00	0	200	380	556	730	1,074	1,246	1,589	2,102
	8.0%	17.0%	24.4%	30.9%	36.8%	47.3%	52.1%	61.2%	73.6%

Note: Table shows NPV_8 $M and IRR.

Figure 22.4 Cumulative Cash Flow

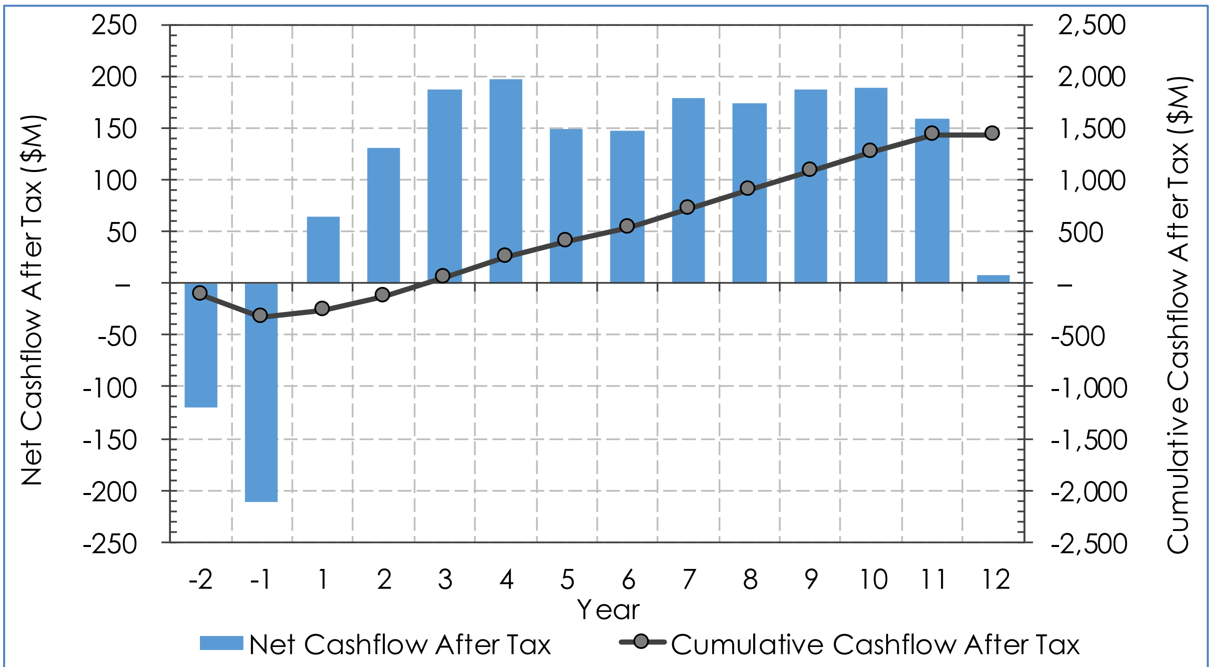


Figure by OreWin, 2017.

Table 22.10 Estimated Cash Flow

Description	Unit	Total	Year													
			-2	-1	1	2	3	4	5	6	7	8	9	10	11	12
Total Gross Revenue	US$M	5,095	–	–	279	420	484	490	463	426	509	507	536	537	444	–
Total Realisation Costs	US$M	1,883	–	–	104	155	177	179	170	157	185	189	200	201	166	–
Net Revenue	US$M	3,212	–	–	175	265	307	311	293	269	324	318	336	336	278	–
Site Operating Costs																
Total Mining	US$M	452	8	29	40	43	40	39	41	37	37	36	35	35	31	–
Processing Zn	US$M	193	–	7	17	18	17	17	17	17	17	17	17	17	15	–
General & Administration	US$M	155	0	11	14	13	13	13	13	13	13	13	13	13	13	–
Total Operating Costs	US$M	800	9	46	71	74	71	70	71	67	67	65	65	65	59	–
Operating Surplus / (Deficit)	US$M	2,412	-9	-46	104	190	236	242	223	202	257	253	271	271	219	–
Capital Costs																
Mine Refurbishment	US$M	17	17	–	–	–	–	–	–	–	–	–	–	–	–	–
Total Mining	US$M	185	24	34	17	26	14	16	22	9	8	7	2	3	4	–
Process & Infrastructure	US$M	73	15	58	–	–	–	–	–	–	–	–	–	–	–	–
Closure	US$M	20	–	–	–	–	–	–	–	–	–	–	–	–	–	20
EPCM	US$M	12	2	9	–	–	–	–	–	–	–	–	–	–	–	–
Owners	US$M	11	2	9	–	–	–	–	–	–	–	–	–	–	–	–
Contingency	US$M	37	7	30	–	–	–	–	–	–	–	–	–	–	–	–
Total Capital	US$M	354	68	139	17	26	14	16	22	9	8	7	2	3	4	20
Cash Flow Before Tax	US$M	2,058	-76	-186	87	165	222	226	201	193	248	246	269	268	215	-20
Federal Income Tax	US$M	509	–	–	1	18.1	24	25	52	46	60	70	76	76	60	–
Cash Flow After Tax	US$M	1,550	-76	-186	86	147	198	201	149	147	188	176	192	192	155	-20
Change in Working Capital	US$M	–	1	4	-19	-11	-6	-1	2	3	-7	1	-2	-0	6	27
Free Cash Flow After Tax	US$M	1,550	-75	-181	68	136	193	200	151	149	181	176	190	192	161	8

22.3 Comparison to Other Projects

Using data for other zinc projects provided by Wood Mackenzie comparisons with the Kipushi 2017 PFS were made for the following results: contained zinc in Measured and Indicated Resource, production, capital intensity, and C1 Cash Costs.

The Kipushi Project Mineral Resource Estimate, 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

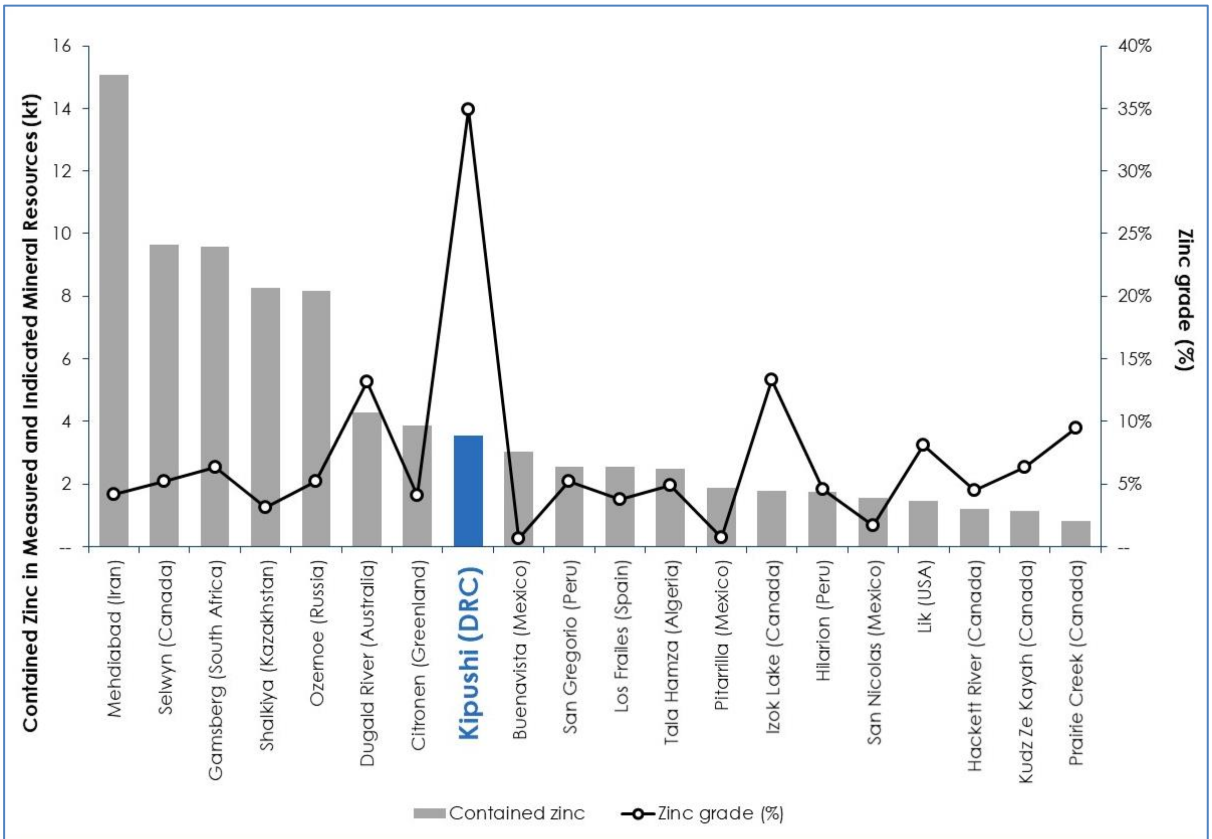


Figure by Wood Mackenzie, 2017.

Life-of-mine average planned zinc concentrate production of 381 ktpa, with a concentrate grade of 59% Zn, is expected to rank the Kipushi Project, once in production, among the world's major zinc mines (Figure 22.6). Based on research by Wood Mackenzie the world's major zinc mines defined as the world's 10 largest zinc mines ranked by forecasted production by 2018.

Figure 22.6 Major Zinc Mines Estimated 2018 Annual Zinc Production and Grade

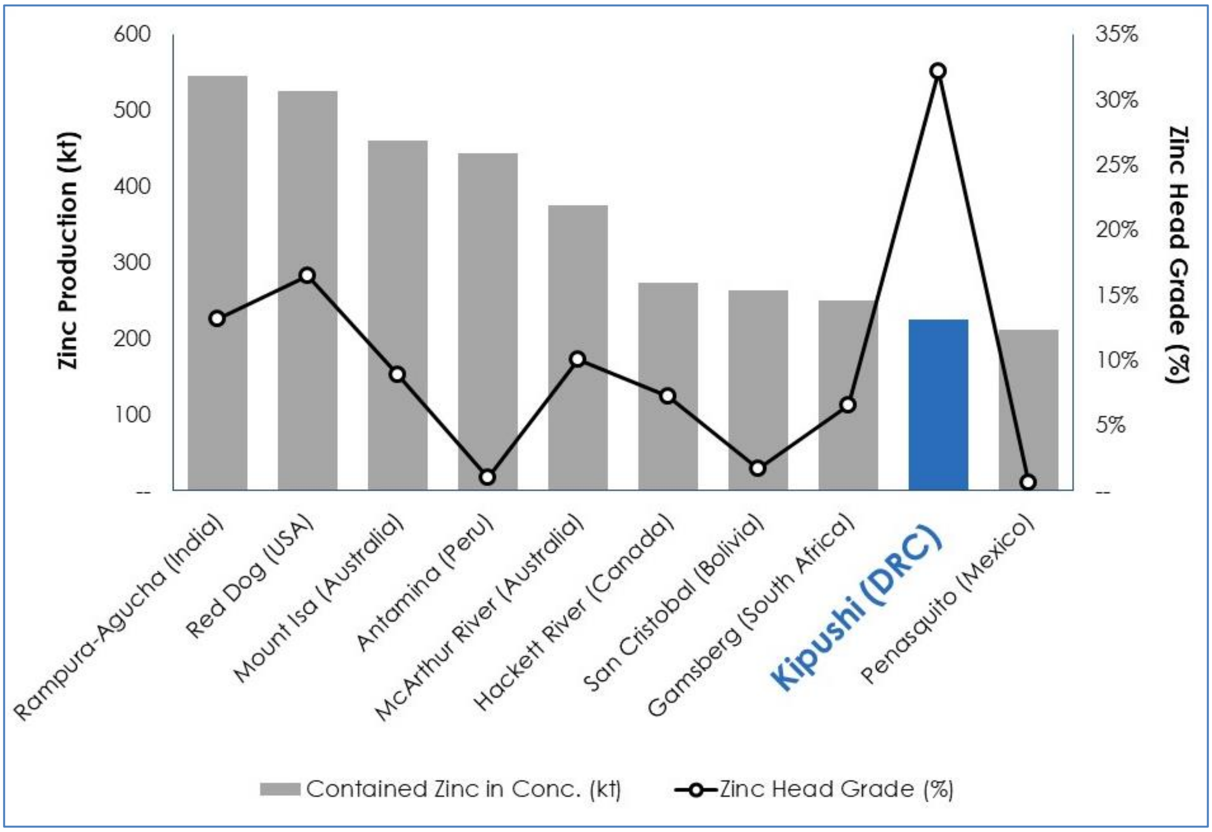


Figure by Wood Mackenzie, 2017.

Kipushi's estimated low capital intensity relative to comparable "probable" and "base case" zinc projects identified by Wood Mackenzie is highlighted in Figure 22.7. The figure uses comparable projects as identified by Wood Mackenzie, based on public disclosure and information gathered in the process of Wood Mackenzie's research.

Figure 22.7 Capital Intensity for Zinc Projects

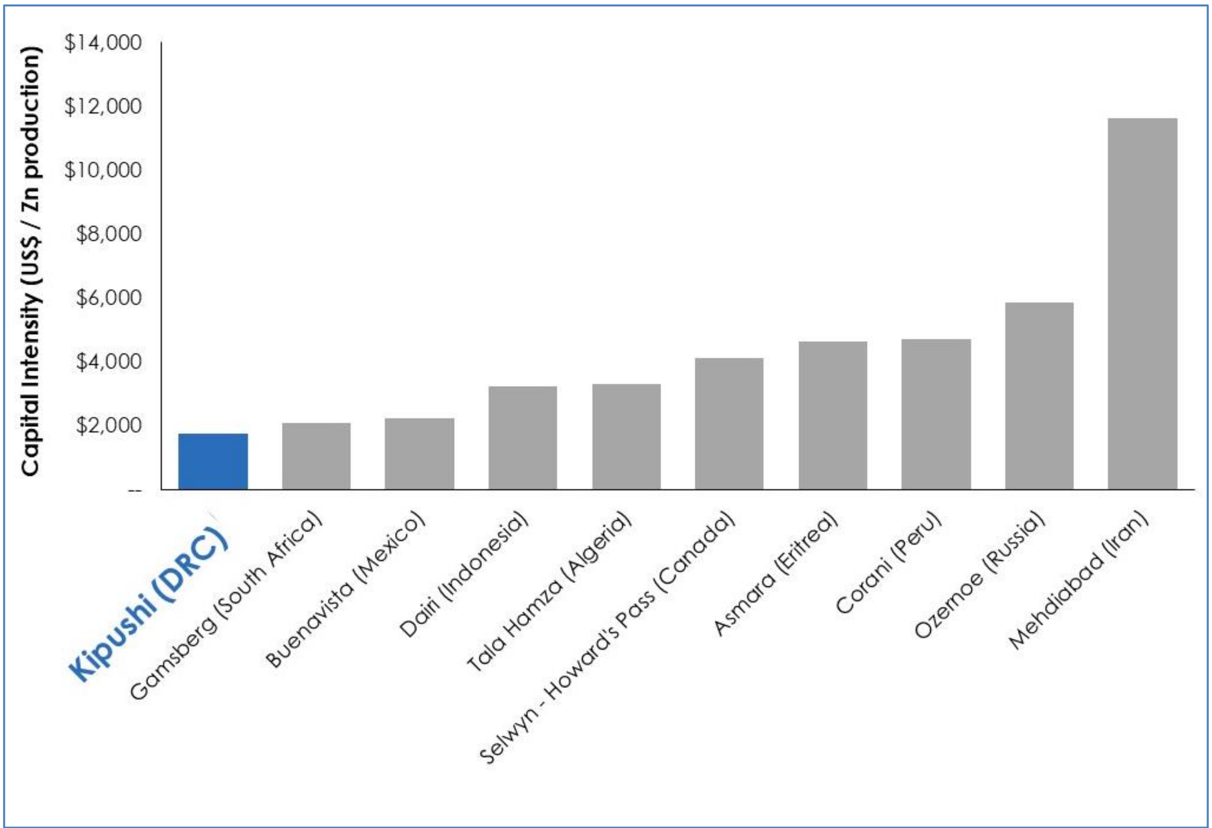


Figure by Wood Mackenzie, 2017.

Based on comparative data from Wood Mackenzie, C1 cash cost of US$0.48/lb of zinc is expected to rank the Kipushi Project, once in production, in the bottom quarter of the 2018 cash cost curve for zinc producers globally. Figure 22.8 represents C1 cash costs which reflect the direct cash costs of producing paid metal incorporating mining, processing and offsite realisation costs having made appropriate allowance for the co-product revenue streams. Based on public disclosure and information gathered in the process of Wood Mackenzie's research.

Figure 22.8 2018 Expected C1 Cash Costs

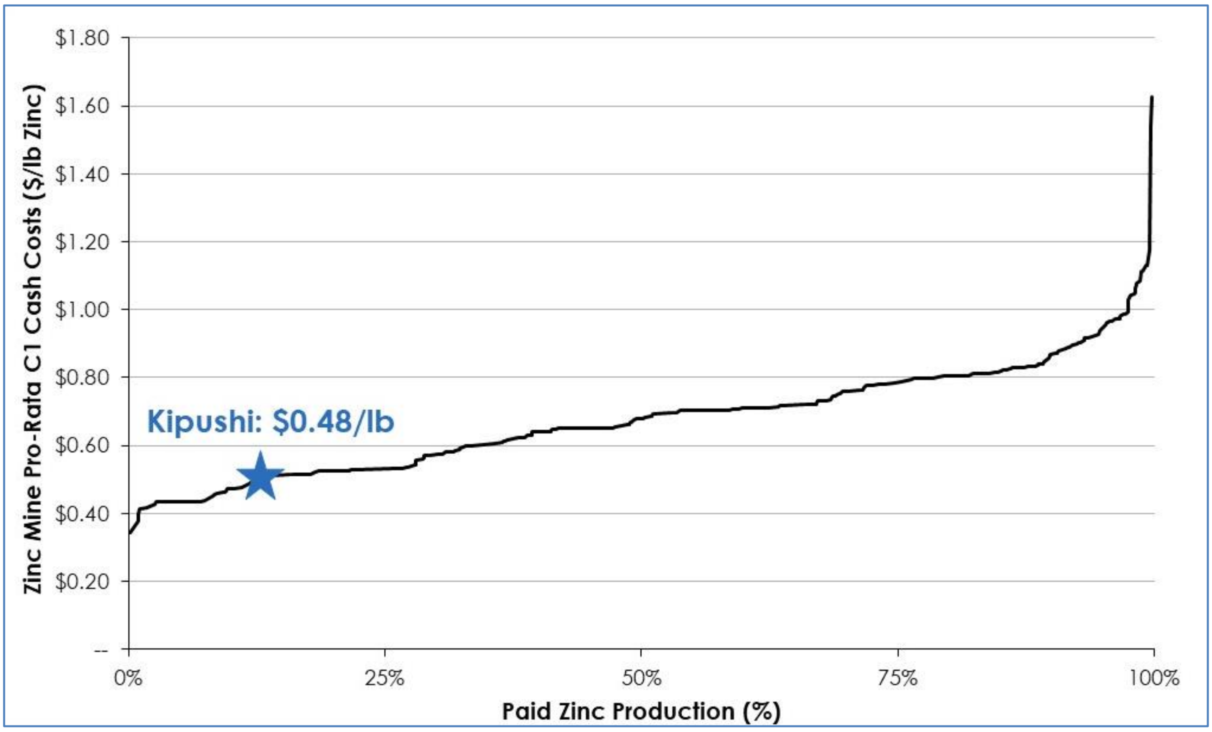


Figure by Wood Mackenzie, 2017.

23 ADJACENT PROPERTIES

This section not used.

24 OTHER RELEVANT DATA AND INFORMATION

This section not used.

25 INTERPRETATION AND CONCLUSIONS

The Kipushi 2019 Resource Update provides an update of the Kipushi Mineral Resource, with the Kipushi Mineral Reserve and the results of the preliminary economic assessment (PEA) from the Kipushi 2017 Prefeasibility Study remaining the same. Aside from the updated Mineral Resource, further study work is currently incomplete and has not determined any results that require material changes to the Kipushi 2017 PFS.

The Kipushi 2017 PFS for the redevelopment of the Kipushi Mine is at a prefeasibility level of study, it has identified a positive business case and it is recommended that the Kipushi Project is advanced to a 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 Project. The key areas for further work are:

25.1 Mineral Resources

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

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

- Assumptions used to generate the data for consideration of reasonable prospects of eventual economic extraction for the Kipushi deposit.
- Metallurgical recovery assumptions in the presence of pyrite- or iron-rich zones.
- Exploitation of the Kipushi Project requires rehabilitation of existing mine infrastructure, building of a new processing facility, and rehabilitation or building of transportation infrastructure. Changes in the assumptions as to operating and capital costs associated with the proposed development may affect the base case cut-off grades selected for the Kipushi Mineral Resource estimates.
- Commodity prices and exchange rates.

25.2 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.

25.3 Mining

- Complete shaft and underground rehabilitation work.
- Additional study work to define the declines, ventilation, and material handling pass systems for FS.
- 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.

25.4 Process

- Flowsheet optimisation tests should be conducted to assess other opportunities including but not limited to direct milling and flotation of ROM ore incorporating cleaner flotation stage in the zinc flotation section; bulk sulphides flotation to reduce reagent consumption, etc.
- Variability testwork program should be conducted to review DMS and flotation plant performance for expected variations in feed concentrations.
- Reviewing the flowsheet to optimise surge capacities, allow bypass of sections and cope with mass pull variations.
- Updating the design with suitable re-assessed ore characteristics, update mine plan and incorporation of thickening and filtration testwork results.
- Reviewing implications for water management and flotation performance associated with mine and tailings water use.
- Identify if the high cost of packaging in 1.8 tonne bags can be practically reduced.

25.5 Infrastructure

- Define the rail option development.
- Progress agreements for rail transportation and engage with the rail contractor.
- Optimise proposed surface infrastrucute layout.
- Evaluate container/bulk shipping with shipping companies.
- Finalize location of tailings storage facility.
- Site survey.

25.6 Marketing

- Investigate customer uptake for container transport.
- Conduct a detailed marketing study and identify customers.

25.7 Environmental and Social

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

25.8 Project Financing

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

26 RECOMMENDATIONS

26.1 Further Assessment

The findings and recommendations of the Kipushi 2017 PFS remain current, and further studies on the Kipushi deposit are in progress but are not yet complete.

The Kipushi 2017 PFS identified a positive business case and recommended that the Kipushi Project advance to a 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 Project.

The results of the Kipushi 2017 PFS suggest that further study should be undertaken. In particular, the investigation of logistics and transport, mining method and processing.

26.2 Drilling

No further drilling is recommended prior to the commencement of mining.

26.3 Underground Mining

The following is a list of mining recommendations for the Project:

- Complete hydrological studies and data evaluation to better determine impacts on underground mining conditions and productivities at Kipushi.
- Drill geotechnical holes to determine ground conditions at each ventilation raise.
- Determine the virgin rock temperature gradient.
- Develop an operating philosophy to optimize waste rock movements.
- Perform a detailed simulation of the underground traffic flow at peak production.
- Conduct a survey of the local workforce to determine available skill levels. The mining productivities and costs have assumed that skilled tradesmen are available to fill the critical mine operational positions.

26.4 Process Plant

The following is a list of process recommendations for the Kipushi deposit:

- Kipushi Corporation SA should develop a reliable and economic measurement method to estimate the zinc mineralogy of samples. This will assist in the prediction of concentrate grades and zinc recoveries. Planned variability testing must proceed and the suitability of the flotation flowsheet must be critically analysed. The most critical unresolved process issue is prediction of zinc concentrate grade and recovery to a level that will support production planning requirements.
- Anomalies in the current Crusher Work Index (CWI) determinations need to be resolved with additional testing of the variability samples. Subsequently, the crusher designs may require updating.

27 REFERENCES

Batumike, M.J., J.L.H. and Kampunzu, A.B. (2007). Lithostratigraphy, basin development, base metals deposits, and regional correlations of the Neoproterozoic Nguba and Kundulungu rock successions, central African Copperbelt. Gondwana Research, 11, p432–447.

Briart, G. (1947). Le gisement de Kipushi. Union Minière du Haut Katanga Geological Department (Likasi, DRC). Unpublished report.

Brooks, T. (2015). Geological setting and mineralisation. Internal report for Ivanhoe Mines.

Brooks, T. (2014). A new look at the Kipushi Cu-Zn-Ge-Ag-Pb deposit, Democratic Republic of Congo. Presented at the GSSA African Exploration Showcase Day on 7 November 2014.

Cailteux, J. (1994). Lithostratigraphy of the Neoproterozoic Shaba-type (Zaire) Roan Supergroup and metallogenesis of associated stratiform mineralisation. Journal of African Earth Sciences, 19, pp.:279–301.

Chabu, M. (2003). Alteration of the host dolomite adjacent to massive zinc mineralisation of the Kipushi Pb-Zn-Cu deposit (Katanga, DRC) and incidences on exploration methods. Third Workshop of IGCP 450: Proterozoic Sediment-hosted Base Metal Deposits of Western Gondwana. Lubumbashi, D.R. Congo. 14–24 July 2003, p138–141.

Chabu M. and Boulegue J. 1992. Barian feldspar and muscovite from the Kipushi Pb-Zn-Cu deposit (Shaba, Zaïre). Canadian Mineralogist 30, p 1143-1152.

Cox, D.P. and Bernstein, L.R. (1986). Descriptive model of Kipushi Cu–Pb–Zn. USGS.

De Magnée, I., and François, A., 1988. The origin of the Kipushi (Cu, Zn, Pb) deposit in direct relation with a Proterozoic salt diapir. Copperbelt of central Africa, Shaba, Republic of Zaïre, in Friedrich, G.H., and Herzig, P.M., eds., Base metal sulfide deposits: Berlin Heidelberg, Springer-Verlag, p. 74–93.

Ehrlich, R.P., and Brady, B.S. (1996). A Review of the Kipushi Mine of é in the Republic of Zaire for America Mineral Fields Inc. Watts, Griffis and McOuat Ltd. Consulting Geologists and Engineers, 12 June 1996, Revised 20 June 1996.

François, A.P., and Cailteux, J., 1981, La Couverture Katangienne entre les Socles de Zilo et de la Kambompo, République du Zaire—région de Kolwezi: Tervuren, Belgique, Musée Royal de l'Afrique Centrale, Annales Serie In 8o, Sciences Géologiques, no. 87, 50 p.

Heijlen, W., Banks, D.A., Muchez, P., Stensgard, B.M. and Yardley, B.W.D. (2008). The nature of mineralising fluids of the Kipushi Zn-Cu deposit, Katanga, Democratic Republic of Congo: Quantitative fluid inclusion analysis using laser ablation ICP-MS and bulk crush – leach methods. Economic Geology, v103, pp1159–1482.

Hubert André-Dumont (2013). Democratic Republic of the Congo. Report prepared by McGuireWoods LLP in Bourassa, M. and Turner J. (2013) Mining in 31 Jurisdictions Worldwide. Published by Getting the Deal Through. Accessed 12 February 2015 at http://www.mcguirewoods.com/news-resources/publications/international/mining-drcongo.pdf.

Intiomale, M.M. and Oosterbosch, R. (1974). Géologie et géochimie du gisement de Kipushi, Zaire. Centenaire de la Société Géologique de Belgique, Gisements stratiformes et Provinces cuprifères, pp123–164.

Intiomale, M.K. (1982). Le Gisement Zn-Pb-Cu de Kipushi (Shaba – Zaire). Etude Geologique et Metallogenique. DSc thesis submitted to Universite Catholique de Louvain. 170p excluding maps and figures.

Ivanhoe Mines (2014). Annual Information Form for the year ended 31 December 2013. 114p.

Ivanhoe Mines (2014a). Ivanhoe Mines' drilling program extends depths of rich zinc and copper zones at the Kipushi Mine in D.R Congo. News Release dated 15 April 2014.

Ivanhoe Mines (2014b). First assay results from Ivanhoe Mines' drilling program confirms high zinc and copper grades at the Kipushi Mine in D.R Congo. News Release dated 14 July 2014.

Ivanhoe Mines (2014c). Additional high-grade zinc and copper assay results returned from drilling program at the Kipushi Mine in D.R. Congo. News Release dated 5 September 2014.

Ivanhoe Mines (2014d). Ivanhoe Mines reports exceptionally high-grade silver, zinc and copper assay results from Big Zinc drillholes at the Kipushi Mine in the Democratic Republic of Congo. News Release dated 24 November 2014.

Ivanhoe Mines (2015). Annual Information Form for the year ended 31 December 2014. 114p.

Ivanhoe Mines (2015a). Ivanhoe Mines reports additional high-grade zinc, copper, silver and germanium drill results from ongoing exploration program at the Kipushi Mine in the Democratic Republic of Congo. News Release dated 17 February 2015.

Ivanhoe Mines (2015b). Ivanhoe Mines reports additional high-grade zinc, copper, silver and germanium drill results from ongoing exploration program at the Kipushi Mine in the Democratic Republic of Congo. News Release dated 13 April 2015.

Jackson, M.P.A., Warin, O.N., Woad, G.M., and Hudec, M.R., 2003, Neoproterozoic allochthonous salt tectonics during the Lufilian orogeny in the Katangan Copperbelt, central Africa: Geological Society of America Bulletin, v. 115, p. 314–330.

Kampunzu, A.B., and Cailteux, J., 1999, Tectonic evolution of the Lufilian arc (central Africa Copper Belt) during Neoproterozoic Pan African orogenesis: Gondwana Research, v. 2, p. 401–421.

Kampunzu, A.B., Cailteux, J.L.H., Kamona, A.F., Intiomale, M.M. and Melcher, F. (2009). Sediment-hosted Zn-Pb-Cu deposits in the Central African Copperbelt. Ore Geology Reviews, 35, p263–297.

Kelly, V., Bennett, J. and Smith, D.J.F. (2012). Kipushi Project, Democratic Republic of Congo. NI 43-101 Technical Report. IMC Group Consulting Limited. Dated 5 September 2012.

Robertson, M. (2013). Kipushi Big Zinc – Orientation Sample Results. Internal report prepared by The MSA Group for Kipushi Corporation SA and dated 13 August 2013. 31p.

Robertson, M. (2014). Kipushi Big Zinc – Results of Core Resampling Programme. Internal report prepared by The MSA Group for Kipushi Corporation SA and dated 29 May 2014. 104p.

Selley, D., Broughton, D., Scott, R., Hitzman, M., Bull, S., Large, R., McGoldrick, P., Croaker, M., Pollington, N., Barra, F., 2005: A New Look at the Geology of the Zambian Copperbelt: in Hedenquist, J.W., Thompson, J.F.H., Goldfarb, R.J., Richards, J.P. (eds.), Economic Geology 100th Anniversary Volume, pp. 965–1,000.

Selley D., Scott R., Emsbo P., Koziy L., Hitzman M.W., Bull S.W., Duffett M., Sebagenzi S., Halpin J. & Broughton D. 2018. Structural Configuration of the Central African Copperbelt: Roles of Evaporites in Structural Evolution, Basin Hydrology, and Ore Location. SEG Special Publications, no. 21, pp. 115–156.

Sketchley, D.A. (2015a). Kipushi Project - QAQC Summary Report. Drillholes: KPU-001 – KPU-087. Period: 1 May 2014 to 1 September 2015. Report prepared for Ivanhoe Mines Ltd and Kipushi Corporation SA and dated 13 May 2015.

Sketchley, D.A. (2015b). Kipushi Mine Project. Drilling Evaluation Program QAQC Check Assay Review. Period ending September 2014. Report prepared for Ivanhoe Mines Ltd and Kipushi Corporation SA covering drillholes KPU001 to KPU025 and dated 30 March 2015.

Sketchley, D.A. (2015c). Kipushi Mine Project. Drilling Evaluation Program QAQC Check Assay Review. Period: September 2014 to April 2015. Report prepared for Ivanhoe Mines Ltd and Kipushi Corporation SA covering drillholes KPU026 to KPU072 and dated 10 August 2015.

Sketchley, D.A. (2018). Kipushi Project – QAQC Report. Drillholes KPU001 – KPU156. Period May 1, 2014 to March 21, 2018. Report prepared for Ivanhoe Mines Ltd and Kipushi Corporation SA, dated 4 July 2018. pp. 46.

Trueman, E.A.G. (1998). Carbonate-hosted Cu±Pb±Zn (Tsumeb or Kipushi Type). In Geological Fieldwork 1997, British Columbia Ministry of Employment and Investment, Paper 1998-1, pp24B1–4.

Tshileo, P.M., Walisumbu, C.K., and Kaluendi, K. (2003). The Kipushi Zn-Pb-Cu Deposit. Third Workshop of IGCP 450: Proterozoic Sediment-hosted Base Metal Deposits of Western Gondwana. Lubumbashi, D.R. Congo. 14–24 July 2003, p210–213.

Turner E.C., Broughton D.W. and Brooks T. 2018. Neoproterozoic Carbonate Lithofacies and Ore Distribution at the Kipushi Cu-Zn Deposit, Democratic Republic of Congo, and Gayna River Zn Camp, Northwest Territories, Canada. Economic Geology, v113, no. 3, p 779-788.

Unrug, R. (1988). Mineralisation controls and source of metals in the Lufilian Fold Belt, Shaba (Zaire), Zambia and Angola. Econ. Geol. 83, p127–128.

Wells, A.P., Scott, N.C. and Hawke, J.N. (2003). Technical Report of the Kipushi Mine Zinc Project in Katanga Province of the Democratic Republic of Congo. IMC Group Consulting Limited. Effective Date 1 August 2003.