OreWin
Independent Mining Consultants

KAMOA COPPER SA

Kamoa-Kakula Project

Kamoa-Kakula Integrated Development Plan 2019

March 2019

Job No. 18006

IMPORTANT NOTICE

This notice is an integral component of the Kamoa-Kakula Integrated Development Plan 2019 (Kamoa-Kakula IDP19) and should be read in its entirety and must accompany every copy made of the Technical Report. The Kamoa-Kakula IDP19 has been prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects.

The Kamoa-Kakula IDP19 has been prepared for Ivanhoe Mines Ltd. (Ivanhoe) by OreWin Pty Ltd (OreWin); Amec Foster Wheeler E&C Services Inc. (a division of Wood PLC); DRA Global (DRA); Stantec Consulting International LLC (Stantec) and SRK Consulting (South Africa) Pty Ltd (SRK) as the Report Contributors. The Kamoa-Kakula IDP19 is based on information and data supplied to the Report Contributors by Ivanhoe and other parties. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the services of the Report Contributors, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in the Kamoa-Kakula IDP19. Each portion of the Kamoa-Kakula IDP19 is intended for use by Ivanhoe subject to the terms and conditions of its contracts with the Report Contributors. Except for the purposes legislated under Canadian provincial and territorial securities law, any other uses of the Kamoa-Kakula IDP19, by any third party, is at that party's sole risk.

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The Kamoa-Kakula IDP19 is intended to be used by Ivanhoe, subject to the terms and conditions of its contract with the Report Contributors. Recognising that Ivanhoe has legal and regulatory obligations, the Report Contributors have consented to the filing of the Kamoa-Kakula IDP19 with Canadian Securities Administrators and its System for Electronic Document Analysis and Retrieval ("SEDAR").

Title Page

Project Name:	Kamoa-Kakula Project
Title:	Kamoa-Kakula Integrated Development Plan 2019
Location:	Lualaba Province Democratic Republic of the Congo
Effective Date of Technical Report:	18 March 2019
Effective Date of Mineral Resources:	Kamoa: 27 November 2017 Kakula: 13 April 2018 Kakula West: 10 November 2018
Effective Date of Drilling Database:	Kamoa: 23 November 2015 Kakula: 26 January 2018 Kakula West: 1 November 2018
Effective Date of Mineral Reserves:	1 February 2019

Qualified Persons:

- Bernard Peters, B. Eng. (Mining), FAusIMM (201743), employed by OreWin as Technical Director - Mining was responsible for: Sections 1.1 to 1.5, 1.15, 1.15.3, 1.15.4, 1.16, 1.16.3, 1.16.4, 1.17, 1.17.2, 1.18, 1.18.1, 1.19, 1.19.1; Section 2; Section 3; Section 4; Section 5; Section 19; Section 20; Sections 21.1, 21.1.3, 21.2, 21.2.1; 21.2.8, 21.3; Section 22; Section 23; Sections 24.1, 24.2, 24.3, 24.4, 24.4.2, 24.5, 24.7; Sections 25.2; Section 26.1; Section 27.
- Dr. Harry Parker, SME Registered Member (2460450), Consulting Mining Geologist and Geostatistician, Amec Foster Wheeler E&C Services Inc. (Amec Foster Wheeler), a Wood plc company (Wood) was responsible for: Sections 1.6 to 1.10, 1.12, 1.13, 1.18.2, 1.19.2; Section 2; Section 3; Section 6; Section 7; Section 8; Section 9; Sections 10.1 to 10.6, 10.9, 10.10; Sections 11.1 to 11.3, 11.5 to 11.12; Section 12; Section 14; Section 25.1; Section 26.2; Section 27.
- Gordon Seibel, SME Registered Member (2894840), Principal Geologist, Amec Foster Wheeler, a Wood company, was responsible for: Sections 1.6 to 1.10, 1.12, 1.13, 1.18.2, 1.19.2; Section 2; Section 3; Section 6; Section 7; Section 8; Section 9; Sections 10.1 to 10.6, 10.9, 10.10; Sections 11.1 to 11.3, 11.5 to 11.12; Section 12; Section 14; Section 25.1; Section 26.2; Section 27.
- William Joughin, FSAIMM (55634), employed by SRK Consulting (South Africa) (Pty) Ltd as Principal Consultant, was responsible for: Section 2; Section 10.7; Section 11.3; Section 16.1; Section 24.4.1
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Signature Page

Project Name:	Kamoa-Kakula Project
Title:	Kamoa-Kakula Integrated Development Plan 2019
Location:	Lualaba Province Democratic Republic of the Congo
Effective Date of Technical Report:	18 March 2019

/s B F Peters

Date of Signing:

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/s H M Parker

Date of Signing:

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Gregory Ruiter Pr. Eng, ECSA (980275), Vice President Studies, DRA Global

TABLE OF CONTENTS

TABLES

FIGURES

1 SUMMARY

1.1 Introduction

The Kamoa-Kakula Integrated Development Plan 2019 (Kamoa-Kakula IDP19) is an independent NI 43-101 Technical Report (the Report) prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for Ivanhoe Mines Ltd. (Ivanhoe) for the Kamoa-Kakula Project (the Project) located in the Democratic Republic of Congo (DRC).

The Project is situated in the Kolwezi District of Lualaba Province, DRC. The Project is located within the Central African Copperbelt, approximately 25 km west of the provincial capital of Kolwezi and about 270 km west of the regional centre of Lubumbashi. The Project includes the Kamoa and Kakula stratiform copper deposits. The declines providing the initial mine access into these deposits are approximately 11 km apart.

Ivanhoe is currently developing twin declines from the north, and a single decline from the south, at the Kakula deposit and has completed the development of twin declines at the Kansoko Mine to provide access to the Kansoko areas of the Kamoa deposit. These sets of declines are approximately 11 km apart. Ongoing drilling of the deposits is continuing to update and expand the Mineral Resources.

The previous Technical Report was the Kamoa-Kakula 2018 Resource Update with an effective date in March 2018, this included a restatement of the Kamoa-Kakula 2017 Development Plan which had been released in a Technical Report with an effective date in March 2018.

1.2 Kamoa-Kakula Integrated Development Plan 2019

The Kamoa-Kakula IDP19 provides updates to the Project Mineral Resources, Mineral Reserves and the studies at Prefeasibility (PFS) and Preliminary Economic Assessment (PEA) stages. The following are the key features of the Kamoa-Kakula IDP19:

- Kakula 2019 PFS (first Mineral Reserve with a plant throughput rate of 6 Mtpa).
- Kamoa 2019 PFS (Mineral Reserve restatement with a plant throughput rate of 6 Mtpa).
- Kamoa-Kakula 2019 PEA (18 Mtpa plant expansion, smelter and five additional mines).
- Separate Kakula and Kakula West Mineral Resource updates.

The Kakula 2019 PFS and Kamoa 2019 PFS include the separate capital and operating costs and assumptions each with its own 6 Mtpa underground mine, processing plant and infrastructure. As such the studies examine two wholly independent operations that do not rely on common infrastructure and would be the subject of separate development decisions. The Kakula 2019 PFS Mineral Reserve is based on the Kakula Mineral Resource stated in IDP19. The details of the Kakula 2019 PFS and Kamoa 2019 PFS are shown in Sections 15 to 22.

The Kamoa-Kakula 2019 PEA analyses a production case with an expansion of the Kakula concentrator processing facilities, and associated infrastructure to 18 Mtpa and includes a smelter and seven separate underground mining operations with associated capital and operating costs. The locations of the seven mines and the boundaries for the PFS and PEA cases are shown in Figure 1.1. The details of the Kamoa-Kakula 2019 PEA are provided in Section 24). The seven mines ranked by their relative values are:

- Kakula Mine (PFS 6 Mtpa).
- Kansoko Mine (PFS 6 Mtpa).
- Kakula West Mine (PEA 6 Mtpa).
- Kamoa Ouest Mine 1 (PEA 6 Mtpa).
- Kansoko Nord Mine 2 (PEA 6 Mtpa).
- Kamoa Centrale Mine 3 (PEA 6 Mtpa).
- Kamoa Nord Mine 4 (PEA 3 Mtpa).

Figure 1.1 Kamoa-Kakula IDP19 Mining Locations

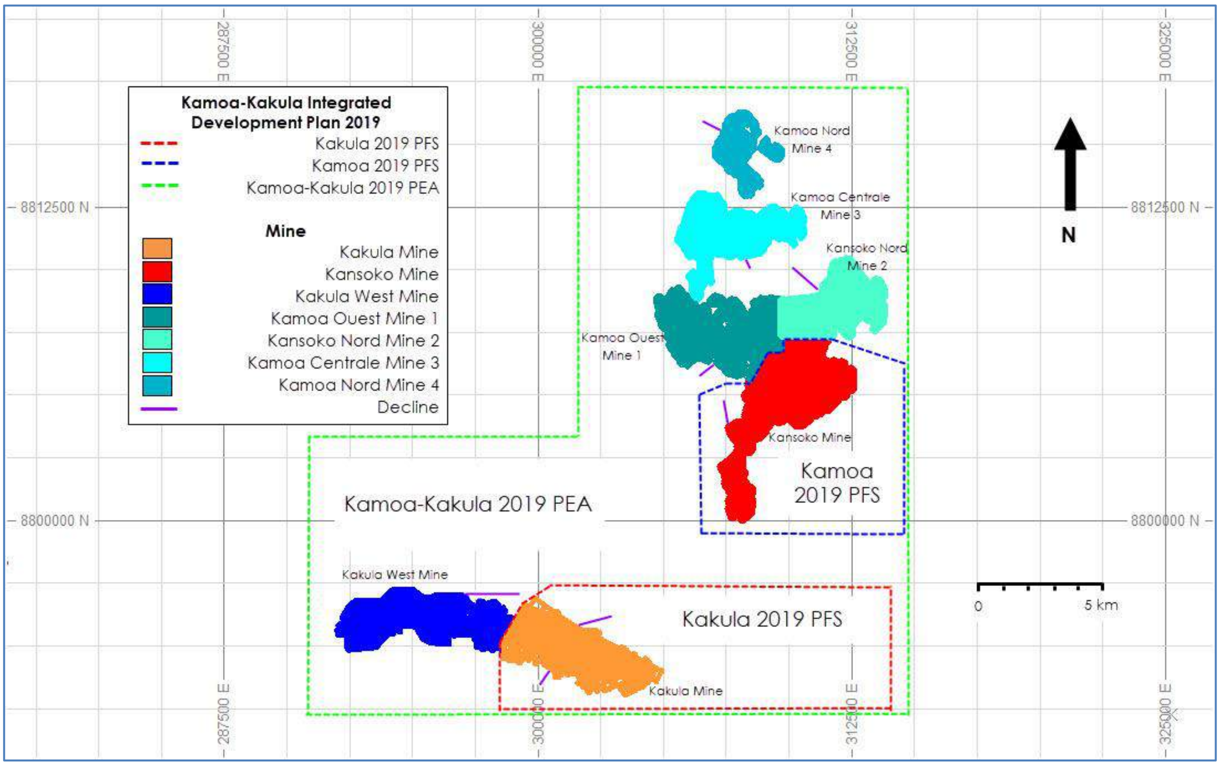


Figure by OreWin, 2019.

The Kamoa-Kakula 2019 PEA is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves – and there is no certainty that the results will be realised. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The Kamoa-Kakula 2019 PEA includes a PEA level study of the whole project including Kakula West and Kamoa North. Kakula West is separated from Kakula by West Scarp Fault and is planned as an independent mine. The Kakula West, Kamoa Ouest, Kansoko Nord, Kamoa Centrale and Kamoa Nord PEA analyses have been prepared using the Mineral Resources stated in the Kamoa-Kakula 2018 Resource Update. Since that time the Kakula and Kakula West Mineral Resources have been updated and the updated Mineral Resources have been stated in Section 14 of the Kamoa-Kakula IDP19.

The potential development scenarios at the Kamoa-Kakula Project include the Kamoa-Kakula IDP19 development scenario shown in Figure 1.2. The Kakula decline development is followed by the development of the stoping panels and construction of the plant. The initial plant capacity of 3 Mtpa is expanded to 6 Mtpa as the Kakula Mine ramps up to full capacity. Following this, the Kansoko Mine is brought into production and the mines continue to ramp up to 12 Mtpa combined by Year 9. The next phase of development described by the Kamoa-Kakula 2019 PEA is from Kakula West to bring total production to 18 Mtpa this is then followed by four new mines at Kamoa North.

The immediate decisions for Ivanhoe and its partners are to determine the sequence for developing the initial operation. A site plan showing the locations of the mines and key infrastructure for Kakula and Kansoko mines is shown in Figure 1.3. The Kamoa-Kakula IDP19 production and economic analysis results are shown in Table 1.1.

Figure 1.2 Kamoa-Kakula IDP19 Long-Term Development Scenario

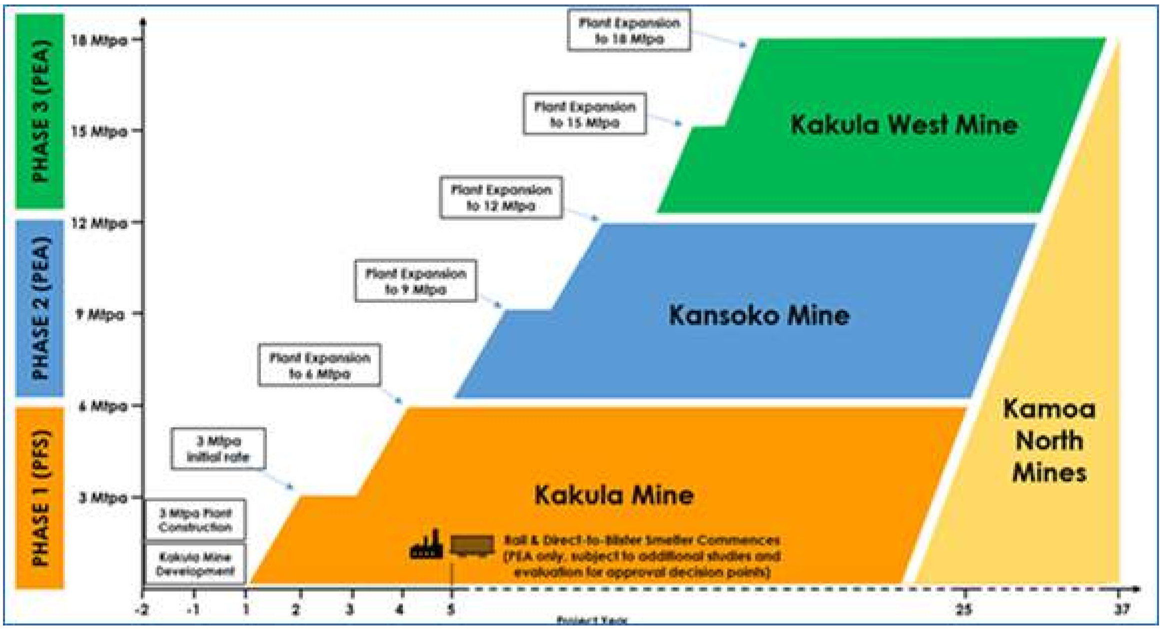


Figure by OreWin, 2019.

Figure 1.3 Kamoa-Kakula IDP19 Site Plan

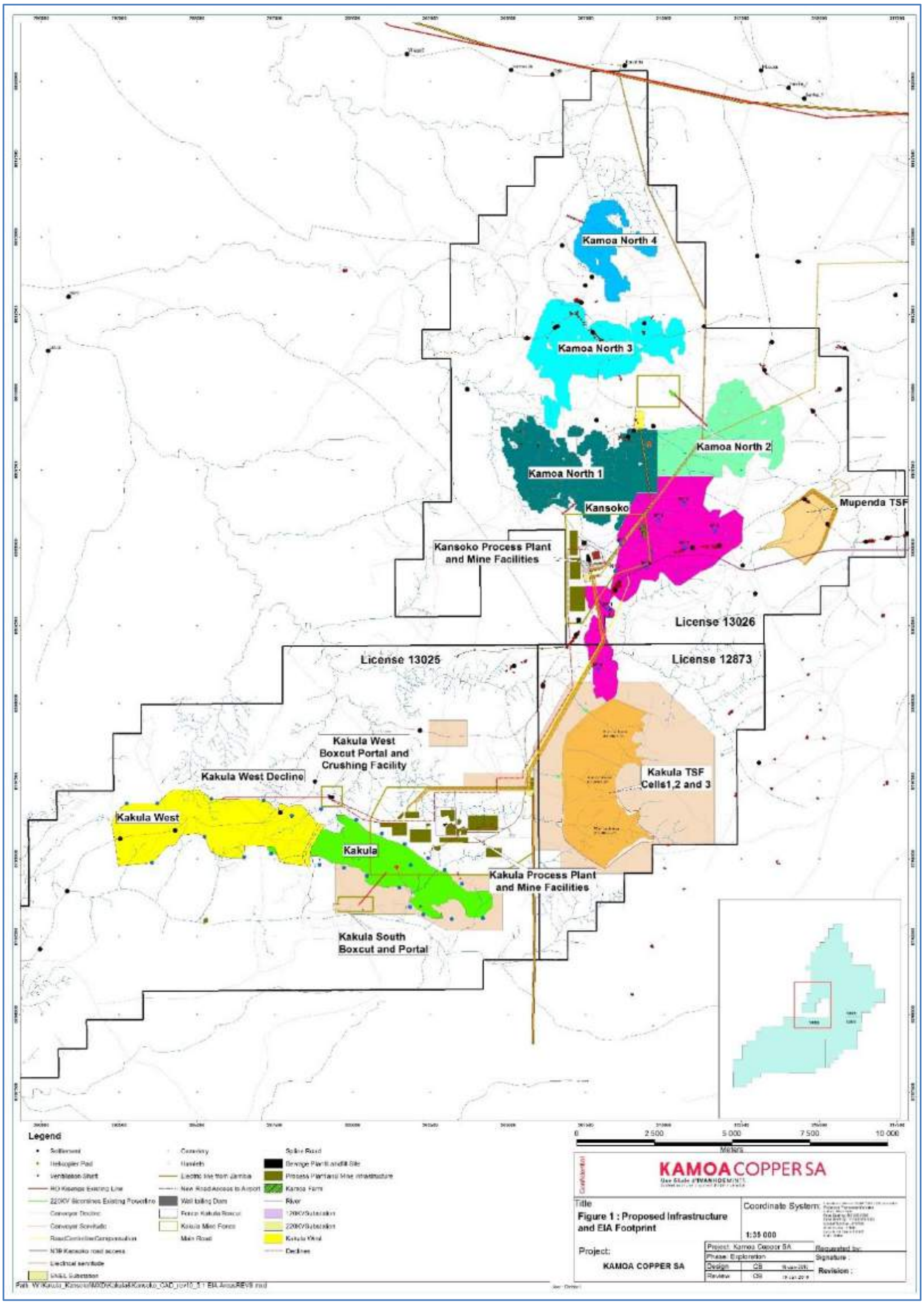


Figure by Kamoa Copper SA, 2019.

Table 1.1 Kamoa-Kakula IDP19 Results Summary

Item	Unit	Kakula 2019 PFS	Kamoa 2019 PFS	Kamoa-Kakula 2019 PEA
Total Processed				
Quantity Milled	kt	119,728	125,182	535,217
Copper Feed Grade	%	5.48	3.81	3.88
Total Concentrate Produced				
Copper Concentrate Produced	kt (dry)	9,776	11,405	39,039
Copper Concentrate - External Smelter	kt (dry)	9,776	11,405	8,491
Copper Concentrate - Internal Smelter	kt (dry)	–	–	30,549
Copper Recovery	%	85.35	87.52	85.12
Copper Concentrate Grade	%	57.32	36.63	45.23
Contained Copper in Conc. - External Smelter	Mlb	12,354	9,211	9,930
Contained Copper in Conc. - External Smelter	kt	5,604	4,178	4,504
Contained Copper in Blister - Internal Smelter	Mlb	–	–	28,559
Contained Copper in Blister - Internal Smelter	kt	–	–	12,954
Peak Annual Recovered Copper Production	kt	360	245	740
10 Year Average				
Copper Concentrate Produced	kt (dry)	508	487	759
Contained Copper in Conc. - External Smelter	kt	291	178	121
Contained Copper in Blister - Internal Smelter	kt	–	–	261
Mine-Site Cash Cost	US$/lb Cu	0.46	0.59	0.63
Total Cash Cost	US$/lb Cu	1.11	1.55	0.93
Key Financial Results				
Peak Funding	US$M	1,099	1,300	1,099
Initial Capital Costs	US$M	1,078	994	1,078
Expansion Capital Costs	US$M	778	299	4,958
Sustaining Capital Cost	US$M	1,295	1,359	10,811
Mine Site Cash Cost	US$/lb Cu	0.59	0.66	0.86
Total Cash Costs After Credits	US$/lb Cu	1.24	1.61	1.10
Site Operating Costs	US$/t Milled	59.44	46.71	61.47
After-Tax NPV8%	US$M	5,440	2,334	10,030
After-Tax IRR	%	46.9	26.7	40.9
Project Payback	Years	2.6	4.7	2.9
Initial Project Life	Years	25	26	37

1.3 Property Description and Location

The Project is situated in the Kolwezi District of Lualaba Province, DRC. The Project is located approximately 25 km west of the provincial capital of Kolwezi, and about 270 km west of the regional centre of Lubumbashi. Ivanhoe discovered the Kamoa copper deposit in 2008, and the high-grade Kakula deposit in 2015.

Access to the Project area from Kolwezi is via unsealed roads to the villages of Kasekelesa and Musokantanda. The road network throughout the Project has been upgraded by Ivanhoe to provide reliable drill and logistical access. A portion of the 1,500 km-long railway line and electric power line from Lubumbashi to the Angolan town of Lobito passes approximately 10 km to the north of the Project area.

The Kolwezi area has distinct dry (May to October) and wet (November to April) seasons. Mining activities in the established mining areas at Kolwezi are operated year-round, and it would be expected that any future mining activities within the Project would also be able to be operated on a year-round basis. Although many companies do not operate during the wet season, Ivanhoe has successfully conducted exploration programmes on a year-round basis over a number of years.

The Project is currently isolated from public infrastructure. Infrastructure on-site is currently limited to support for exploration programs, and the ongoing initial mine development activities in the Kamoa and Kakula deposit areas. Exploitation of the Kamoa and Kakula deposits will require building a greenfields project with attendant infrastructure.

1.4 Mineral and Surface Rights, Royalties, and Agreements

The Project consists of the Kamoa Exploitation Licences (exploitation permits 12873, 13025, and 13026 which cover an area of 397.4 km^2) and one exploration licence (exploration permit 703 which covers an area of 12.74 km^2). The Kamoa Exploitation Licences, approved 20 August 2012, grant Ivanhoe the right to explore for, develop and exploit copper and other minerals, for an initial 30-year term, expiring 19 August 2042. The permits can then be extended for 15-year periods, until the end of the mine's life.

Title to the Project resides with Kamoa Copper SA, a subsidiary of Kamoa Holding Limited (Kamoa Holding), which is the holder of the Kamoa Exploitation Licences.

Ivanhoe owns a 49.5% share interest in Kamoa Holding, an Ivanhoe-Zijin subsidiary that presently owns 80% of the Kamoa-Kakula Project. Zijin Mining Group Co. Ltd. (Zijin) owns a 49.5% share interest in Kamoa Holding, which it acquired from Ivanhoe in December 2015 for an aggregate cash consideration of US$412 million. The remaining 1% interest in Kamoa Holding is held by privately-owned Crystal River Global Limited. A 5%, non-dilutable interest in Kamoa Copper SA was transferred to the government of the DRC following the shareholders' general meeting dated 11 September 2012, for no consideration, pursuant to the DRC Mining Code.

On 11 November 2016, the DRC Minister of Mines and Minister of Portfolio, Ivanhoe and Zijin signed an agreement that transfers an additional 15% interest in the Kamoa-Kakula Project to the DRC government, increasing its total stake in the Project to 20%. As a result of the transaction, Ivanhoe and Zijin each hold an indirect 39.6% interest in the Kamoa-Kakula Project while Crystal River Global Limited holds an indirect 0.8% interest and the DRC Government holds a direct 20% interest in the Kamoa-Kakula Project.

Land access for the exploration programmes completed to date has typically been negotiated without problems. Where compensation has been required for exploration activities, compensation has followed International Finance Corp (IFC)/World Bank Guidelines in all cases.

Holders of mining rights are subject to taxes, customs and levies defined in the 2002 Mining Code for all mining activities carried out by the holder in the DRC.

On 9 March 2018, Law No. 18/001 amending the 2002 Mining Code was promulgated (the 2018 Mining Code). The revised regulatory and fiscal regime, which is applicable from March 2018, does not take into account the stability provisions granted to holders of existing mining licenses and remains a point of contention between the mining industry and the DRC Government.

For the purpose of this report, the economic analysis is based on the 2018 Mining Code. According to the 2018 Mining Code, a company holding a mining exploitation licence is subject to payment of mining royalties. The royalty is due upon the sale of the product and is calculated at 3.5% of the value of metal sold, which is defined as 65% for copper in concentrate (31 – 60% copper content) and 95% for copper in blister (90 – 98% copper content)..

1.5 Project Ownership

On 8 December 2015, Ivanhoe and Zijin Mining Group Co., Ltd. (Zijin) closed an agreement to co-develop the Project. Under terms of agreements, Zijin through its subsidiary, Gold Mountains (H.K.) International Mining Company Limited owns 49.5% share interest in Kamoa Holding Limited (Kamoa Holding). Crystal River Global Limited is a private company that owns 1% of Kamoa Holding. At the time of the agreement, Kamoa Holding owned 95% of the Project. The relationship between Ivanhoe Mines, Zijin, and Crystal River Global Limited will be governed by a Shareholder, Governance and Option Agreement (SGOA). The SGOA provides, among other things, that all key decisions regarding the development and operation of the Project will be made by Kamoa Holding's Board of Directors. A 5%, non-dilutable interest in Kamoa Holding was transferred to the government of the DRC on 11 September 2012, for no consideration, pursuant to the DRC Mining Code.

On 11 November 2016, the DRC Minister of Mines and Minister of Portfolio, Ivanhoe and Zijin Mining Group Co., Ltd., signed an agreement that transfers an additional 15% interest in the Kamoa-Kakula Project to the DRC government, increasing its total stake in the project to 20%. As a result of the transaction, Ivanhoe Mines and Zijin each hold an indirect 39.6% interest in the Kamoa-Kakula Project while Crystal River Global Limited holds an indirect 0.8% interest, and the DRC Government holds a direct 20% interest in the Kamoa-Kakula Project.

1.6 Geology and Mineralisation

The mineralisation identified to date within the Project is typical of sediment-hosted stratiform copper deposits. The Kamoa-Kakula mineralisation, however, is unusual in that it is hosted at the base of the Grand Conglomerat, which is stratigraphically higher than the majority of Copperbelt deposits which are typically hosted by dolomitic rocks of the Mines Subgroup.

The regional geology comprises sedimentary rocks of the 840–535 Ma Katangan basin, an intracratonic rift which developed on Paleoproterozoic composite basement rocks. The metasedimentary rocks that host the Central African Copperbelt mineralisation form a sequence known as the Katanga Supergroup, comprising the Roan, Lower Kundelungu, and Upper Kundelungu Groups.

Significant structural complexity evident in the DRC portion of the Copperbelt, particularly evident in the neighbouring Kolwezi district, is not developed at Kamoa-Kakula, which has a far simpler structural configuration similar in style to the southern Congolese and Zambian portions of the Copperbelt At Kamoa-Kakula, the sandstones and siltstones of the Mwashya Group form the oxidised lower strata, with the overlying pyritic diamictite and interbedded siltstone-sandstones of the Lower Kundulungu Group forming the reduced host rock. Whilst likely of glacial origin, the diamictites on the Project are interpreted to be the product of debris flows into a rapidly subsiding basin.

At the Kamoa deposit, the mineralised stratigraphic sequence at the base of the diamictite comprises several interbedded units which appear to control copper mineralisation. These units are, from bottom upward, clast-rich diamictite (Ki1.1.1.1), sandstone and siltstone (Ki1.1.1.2), and clast-poor diamictite (Ki1.1.1.3). The lowermost clast-rich diamictite (Ki1.1.1.1) unit generally hosts lower-grade (<0.5% TCu) mineralisation. Most of the higher-grade mineralisation occurs within the clast-poor (Ki1.1.1.3) unit, or in the sandstone and-siltstone (Ki1.1.1.2) interbeds that are locally present between the clast-rich (Ki1.1.1.1) and clast-poor (Ki1.1.1.3) diamictites. Hypogene mineralisation is characterised by chalcopyrite and bornite-dominant zones. There is significant pyrite mineralisation in the Kamoa Pyritic Siltstone (KPS) above the mineralised horizon that could possibly be exploited to produce pyrite concentrates for sulphuric acid production.

At the Kakula deposit, a deeper basinal setting has resulted in significant thickening of the diamictite basal units with the development of a number of interbedded siltstone units. Mineralisation is concentrated within a basal siltstone layer occurring just above the Roan (R4.2) contact. From the base of mineralisation upward, the hypogene copper sulphides in the mineralised sequence are zoned with chalcocite (Cu_2S), bornite (Cu_5FeS_4) and chalcopyrite ($CuFeS_2$), with chalcocite being the dominant mineral.

Copper mineralisation comprises three distinct styles: supergene, hypogene, and mixed mineralisation. Near the surface adjacent to the domes, the diamictites have been leached, resulting in localised zones of copper oxides and secondary copper sulphide enrichment down-dip in the supergene zones. Although high-grade, these supergene zones are relatively narrow and localised. Hypogene mineralisation forms the dominant mineralisation style. Hypogene mineralisation occurs at depths as shallow as 30 m. All three styles of mineralisation occur at Kamoa; at Kakula all of the mineralisation occurs well below the surface and is hypogene.

1.7 Exploration

Although exploration was undertaken by the Tenke Fungurume Consortium between 1971 and 1975, and localised regional stream-sediment sampling may have occurred in the current Project area, no information on sample locations is available for any sampling that may have occurred within the confines of the current Project.

Work completed by Ivanhoe and third-party contractors on the Project has included geological mapping, geochemical sampling, an airborne geophysical survey, reverse circulation (RC), and core drilling, and petrographic studies.

Exploration activities at the Kamoa-Kakula Project are being augmented by ongoing geophysical exploration programmes. A 3,100 km, airborne gravity survey, covering 2,000 km^2 of the Western Foreland area (including Kamoa-Kakula) was recently completed, and the data are being processed. In addition, four 2D seismic lines have been completed. The seismic survey aimed to position the top of the Roan, interpret broad-scale basin architecture and locate both known and unknown growth and younger brittle structures.

Integration of the geophysical programme results with the Kamoa-Kakula team's existing geological models will allow fine-tuning of exploration targeting within the highly prospective Kamoa-Kakula exploitation licence area.

In the opinion of the Amec Foster Wheeler Qualified Persons (QPs), the exploration programmes completed to date are appropriate to the style of the Kamoa and Kakula deposits. The provisional research work that has been undertaken supports Ivanhoe's deposit genetic and affinity interpretations for the Project area. The Project area remains prospective for additional discoveries of base-metal mineralisation around known dome complexes. Anomalies generated by geochemical, geophysical, and drill programmes to date support additional work on the Project area.

1.8 Drilling

The drillhole database used for the Kamoa resource estimate was closed on 23 November 2015. The resource model for Kamoa was updated as of 27 November 2017. The resource model for Kakula is divided into two portions, split along the West Scarp Fault. The drillhole database used for the Kakula resource estimate (east of the West Scarp Fault) was closed on 26 January 2018. and the resource model was completed as of 13 April 2018. The drillhole database used for the Kakula West resource estimate (west of the West Scarp Fault) was closed on 1 November 2018, and the model was completed as of 10 November 2018.

Coreholes have been used for geological modelling, and those occurring within the mining lease and in areas of mineralisation (drillholes on the Kamoa, Makalu and Kakula domes are excluded) have been used for resource estimation.

As at 1 March 2019, there were 1,853 coreholes drilled within the Kamoa-Kakula Project. The November 2017 Kamoa Mineral Resource estimate used 776 drillholes. Included in the 776 drillholes were 16 twin holes (where the spacing between drillholes is <25 m) and six wedge holes. Although a far greater number of holes have been wedged, the wedges have typically been used in their entirety for metallurgical testing and have thus not been sampled for Mineral Resource estimation purposes. In these cases, only the parent hole is used during Mineral Resource estimation. The Kakula Mineral Resource estimate used 155 drillhole intercepts (one intercept per drillhole). The Kakula West Mineral Resource estimate used 168 drillhole intercepts (one intercept per drillhole).

The 754 holes not included in either the November 2017 Kamoa, or the February 2018 Kakula estimate, or the November 2018 Kakula West estimate were excluded because they were abandoned, unmineralised holes in the dome areas, unsampled metallurgical, civil geotechnical or hydrological drillholes, or were drilled after the closure of the databases. Subsequent to the closure of the database for the Kamoa Mineral Resource estimate (23 November 2015), 110 drillholes have been completed inside of the modelled area at Kamoa. An additional 20 drillholes at Kakula and 28 drillholes at Kakula West have been completed after the closure of the databases on 26 January 2018 and 1 November 2018 respectively. Standard geological logging methods, sampling conventions, and geological codes have been established for the Project. Geotechnical logging has been undertaken on the majority of the drill cores. Kamoa core recovery in the mineralised units ranges from 0% to 100% and averages 95%. Intervals in the database with 0% recovery likely indicate missing data, as logging does not indicate poor recovery. Visual inspection of the Kamoa core by the Amec Foster Wheeler QPs documented the core recovery to be excellent. All completed holes are surveyed by an independent professional surveyor SD Geomatique using a differential GPS which is accurate to within 20 mm.

The Kakula drillholes collars have been surveyed by SD Geomatique and E.M.K. Construction SARL. As of 26 January 2018, there were no outstanding collar surveys east of the West Scarp Fault being used in the Kakula Mineral Resource estimate. As of 1 November 2018, there was one outstanding collar survey (DKMC_DD1386) west of the West Scarp Fault being used in the Kakula West Mineral Resource estimate. Visual inspection of the Kakula core by the Amec Foster Wheeler QPs documented the core recovery to be excellent.

In the opinion of the Amec Foster Wheeler QPs, the quantity and quality of the lithological collar, and downhole survey data collected in the core drill programmes are sufficient to support Mineral Resource estimation at Kamoa and Kakula.

1.9 Sample Preparation, Analyses, and Security

Pre-February 2010, determination of the sample intervals took into account lithological and alteration boundaries, with core generally sampled and assayed from 4 m above the first presence of mineralisation on nominal 1 m intervals to the end of the hole, which is generally 5 m below the Ki1.1/R4.2 contact. In addition, most intervals with visual estimates of >0.1% Cu were sampled at 1.5 m intervals or less.

From February 2010 through July 2014, the Kamoa Pyritic Siltstone (KPS, Ki1.1.2) and mineralised basal diamictite were sampled on nominal 1 m sample intervals (dependent on geological controls). The KPS was sampled every 1 m, and composites were made over 3 m for analytical purposes. A 3 m shoulder is sampled above the first visible sign of copper mineralisation in each drillhole.

Starting in August 2014, whole core is logged by the geologist on major lithological intervals, until they arrive at mineralised material or at a "zone of interest" (ZI) such as a lithology that is conventionally sampled (e.g. the Kamoa Pyritic Siltstone). The ZI is logged on sampling intervals, typically 1 m intervals (dependent on geological controls). Within any zone of interest, the geologist highlights material that is either mineralised or material expected to be mineralised and that could potentially support a Mineral Resource estimate. This is highlighted as "zone of assay" (ZA) and is extended to 3 m above and below the first sign of visible mineralisation.

Independent laboratories have been used for primary sample analysis, Genalysis Laboratory Services Pty. Ltd. (Genalysis, from 2007 part of the Intertek Minerals Group), and Ultra Trace Geoanalytical Laboratory (Ultra Trace, from 2008 owned and operated by the Bureau Veritas Group). Both laboratories are located in Perth, Western Australia, and both have ISO 17025 accreditation.

ALS of Vancouver, British Columbia, acted as the independent check laboratory for drill core samples from part of the 2009 programme and for 2010 through 2018 drilling. ALS is ISO:9001:2008 registered and ISO:17025-accredited.

Sawn drill core is sampled on 1 m intervals, or shorter intervals where necessary, to honour geological contacts. The sawn core is then crushed to nominal 2 mm using jaw crushers. A quarter split (500 g to 1,000 g) is pulverised to >90% -75 µm, using the LM2 puck and bowl pulverisers. The remaining coarse reject material is retained. A 100 g split is sent for assay; three 50 g samples are kept as government witness samples, one 30 g sample is split for Niton (X-ray fluorescence or XRF) analysis, and approximately 80 g of pulp is retained as a reference sample. Certified reference materials (CRMs) and blanks are included with the sample submissions.

Analytical methods have changed over the Project duration. Samples typically are analysed for Cu, Fe, As, and S. A suite of additional elements was requested, in particular, during the early drilling phases at Kamoa. Acid-soluble copper (ASCu) assays have been primarily undertaken at Kamoa since 2010. Very few (249 out of 6,640) samples from holes drilled prior to 2010 have ASCu assays.

Ivanhoe has discontinued ASCu analysis at Kakula. No ASCu results exist for drillholes DKMC_DD1024, DKMC_DD1025, DKMC_DD1031, and all drillholes from DKMC_DD1033 onward. The discontinuation results from all the mineralisation at Kakula being considered to be hypogene.

In the opinion of the Amec Foster Wheeler QPs, the sampling methods are acceptable, are consistent with industry-standard practices, and are adequate for Mineral Resource and Mineral Reserve estimation, and suitable for mine planning purposes.

1.10 Data Verification

Amec Foster Wheeler reviewed the sample chain of custody, quality assurance and control (QA/QC) procedures, and qualifications of analytical laboratories. Amec Foster Wheeler is of the opinion that the procedures and QA/QC control are acceptable to support Mineral Resource estimation. Amec Foster Wheeler also audited the assay database, core logging, and geological interpretations on a number of occasions between 2009 and 2018 and has found no material issues with the data as a result of these audits. Independent witness sampling and assaying programs conducted by Amec Foster Wheeler were found to be consistent with Ivanhoe's original sampling and assaying.

In the opinion of the Amec Foster Wheeler QPs, the data verification programmes undertaken on the data collected from the Project support the geological interpretations. The analytical and database quality and the data collected can support Mineral Resource estimation.

1.11 Metallurgical Testwork and Concentrator Design

1.11.1 Kakula

A number of preliminary Kakula samples were composited and tested since 2016 at both Zijin and XPS laboratories. Initially these material were conducted using the circuit developed during the 2016 Kamoa prefeasibility study. Because of the higher-grade nature of the Kakula material, very high concentrate grade were achieved from both laboratories with copper content in excess of 50% at reasonable high recoveries >86%.

In 2018, as part of the Kakula 2019 PFS, further development test work was conducted on the Kakula material, by XPS. The Kakula 2019 PFS master composite sample, grading 6.1% copper, produced a copper recovery of 86% at a concentrate grade of 57% copper, applying the the recently developed flow sheet specifically suited for the predominately chalcocite rich Kakula material. These results were reproduced consistently at XPS during benchscale sample productions for downstream tests as well as at Zijin laboratory using the same sample and test conditions.

Preliminary variability testworlk on the frozen Kakula 2019 PFS circuit were conducted using ten samples from Kakula with varying head grade from 2.6 to 9.2% copper. These samples perfomed in line with the feed mineralogy and demonstrated that the developed Kakula circuit can handle the variation in feed composition.

Average arsenic levels in the concentrate were measured to be approximately 0.02%, which is significantly lower than the limit of 0.5% imposed by Chinese smelters. Low arsenic levels in concentrate are expected to attract a premium from copper-concentrate traders.

XPS also conducted mineralogical analysis and duplicate float tests using a preliminary Kakula West sample, grading 3.2% copper through the developed Kakula circuit. The mineralogy work indicated that the Kakula West material was high in chalcocite, and had higher levels of chalcopyrite compared to the Kakula material. The average grain size of the Kakula West copper sulphide minerals was noted as similar to the Kansoko material, which are slightly coarser than the Kakula 2019 PFS sample. The Kakula West sample performed very well and achieved a final copper recovery of 86% at a concentrate grade of 54% copper. This test work indicated that the Kakula and Kakula West material could be processed in a common concentrator.

The Kakula concentrator would be constructed in a phased approach with two three Mtpa modules as the mining operations ramp-up to full production of 6 Mtpa.

The Kakula concentrator design incorporates a run-of-mine stockpile, followed by two stages of screening and crushing on surface. The crushed material with a size distribution of 80% passing (or P_{80}) eight millimetres (mm), is fed into a two-stage ball-milling circuit for further size reduction to a target grind size 80% passing 53 micrometres (µm). The milled slurry will be passed through a rougher and scavenger flotation. The high-grade, or fast-floating rougher concentrate, and medium-grade, or slow-floating scavenger concentrate, will be collected separately. The rougher concentrate is upgraded in the low entrainment high-grade cleaner stage to produce a high-grade concentrate. The medium-grade scavenger concentrate together with the tailings from the rougher cleaning stage and the tailings from the scavenger recleaner stage will be combined and further upgraded in the scavenger cleaner circuit. The concentrate produced from the scavenger cleaner circuit, representing roughly 12% of the mill feed, will be re-ground to a P_{80} of 10 µm prior to final cleaning in the low entrainment scavenger recleaner stage. The scavenger recleaner concentrate then will be combined with the high-grade cleaner concentrate to form the final concentrate. The final concentrate will be thickened before being pumped to the concentrate filter where the filter cake will then be bagged for shipment to market.

1.11.2 Kamoa

Between 2010 and 2015 a series of metallurgical testwork programs, defined as Phases 1 to 5, were completed on Kamoa drill core sample and focussed on metallurgical characterisation and flow sheet development for processing the hypogene and supergene material. During this period the orebody was expanded, leading to major changes to mine schedules and associated processing schedules. Given that the new schedules indicated that the supergene mineralisation accounted for less than 10% of the oredoby, the focus shifted to the hypogene ores. These campaigns provided input to the development of a MF2 type flow sheet and the necessary metallurgical understanding to support the 2012 PEA and subsequent Technical Reports.

In preparation for the Kamoa 2016 PFS and the increased capacity Kamoa 2017 PFS, the Phase 6 samples were selected and the associated metallurgical evaluation was conducted during 2014–2015 at Xstrata Process Support (XPS) Laboratories. The Phase 6 samples best represent ores to be processed according to the early years of the Kamoa 2019 PFS mine schedule. It is noted that many of the Phase 2 and Phase 3 samples are relevant to the current Kamoa 2019 PFS mine schedule. The Phase 6 campaign developed the IFS4a flow sheet which was confirmed as the final flow sheet for Kansoko, specifically tailored to the fine-grained nature of the material.

The pervasive presence of ultrafine copper sulphides in all Kamoa ore samples leads to strong recovery of silica through attachment with these sulphides. This, in turn, has led to silica rejection issues in final concentrate production, which is mitigated to a large degree by 10 µm regrinding of middling and scavenging streams. The most recent testwork at two independent laboratories has consistently achieved silica levels in the range 14 to 15% SiO_2 and has provided confidence that this level of silica rejection, at a minimum, will be achievable in operations.

The prediction of copper recovery from hypogene ore is reasonable, but the prediction for supergene samples applicable to Kansoko is currently inadequate. An improved method of supergene recovery prediction for Kamoa mineralisation is necessary. It should be noted that the lack of supergene ore in Kakula makes this concern minor for that deposit.

The Kakula concentrator design is similar to Kamoa circuit except that the regrind circuit is moved to scavenger cleaner concentrate thus reducing the regrind feed to12% of mill feed; and low entrainment flotaticion cells are incorporated for HG cleaning and scavenger recleaning stages.

1.12 Mineral Resource Estimates

Mineral Resources for the Kamoa deposited have not been updated and are the same as those reported in the 10 January 2018 Kamoa-Kakula 2017 Development Plan. The Kakula model was updated and documented in this Report. The Kakula Mineral Resource model is separated into two sections, with the West Scarp Fault forming the boundary. The model for Kakula West (that portion west of the West Scarp Fault), was updated incorporating significant additional drilling up to 1 November 2018. The Kakula model (that portion east of the West Scarp Fault) is based on drilling up to 26 January 2018 (the same dataset used for the previous Kakula model) but the modeling method was updated from a two-dimensional (2D) to a three-dimensional (3D) estimation methodology. All Kamoa and Kakula resource models are now based on the same general 3D estimation methodology. The location of the resource models are shown in relation to the context of the broader Project in Figure 1.4 and a more detailed outline of resources models with the location of the drillholes is shown in Figure 1.5).

Figure 1.4 Kamoa-Kakula Exploitation Licence, Showing the Kamoa, Kakula, and Kakula West Mineral Resource Areas, Kamoa North Exploration Area and Exploration Targets

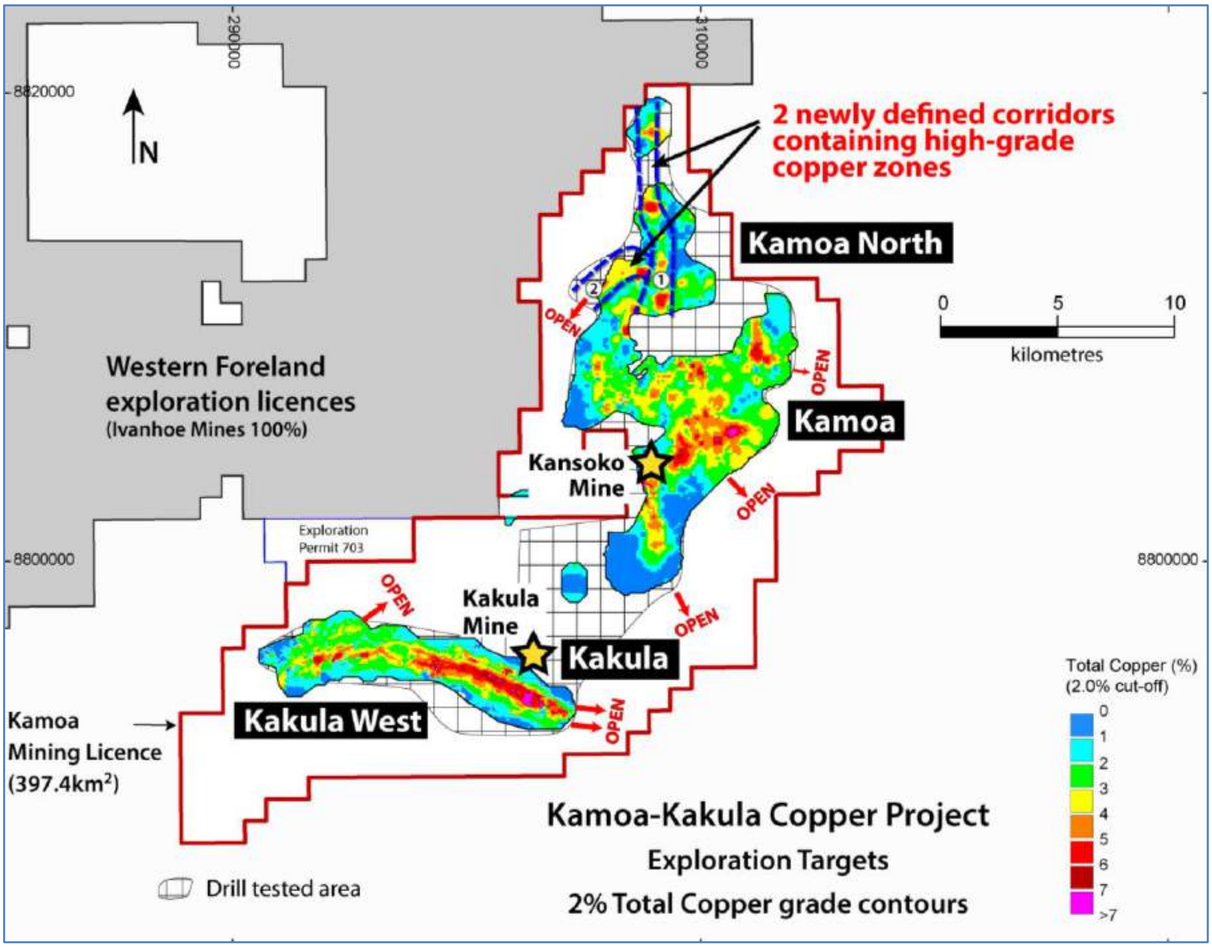


Figure by Kamoa Copper SA, 2019.

Figure 1.5 Location Plan Showing the Outlines of the Kamoa-Kakula Copper Project Indicated and Inferred Mineral Resources

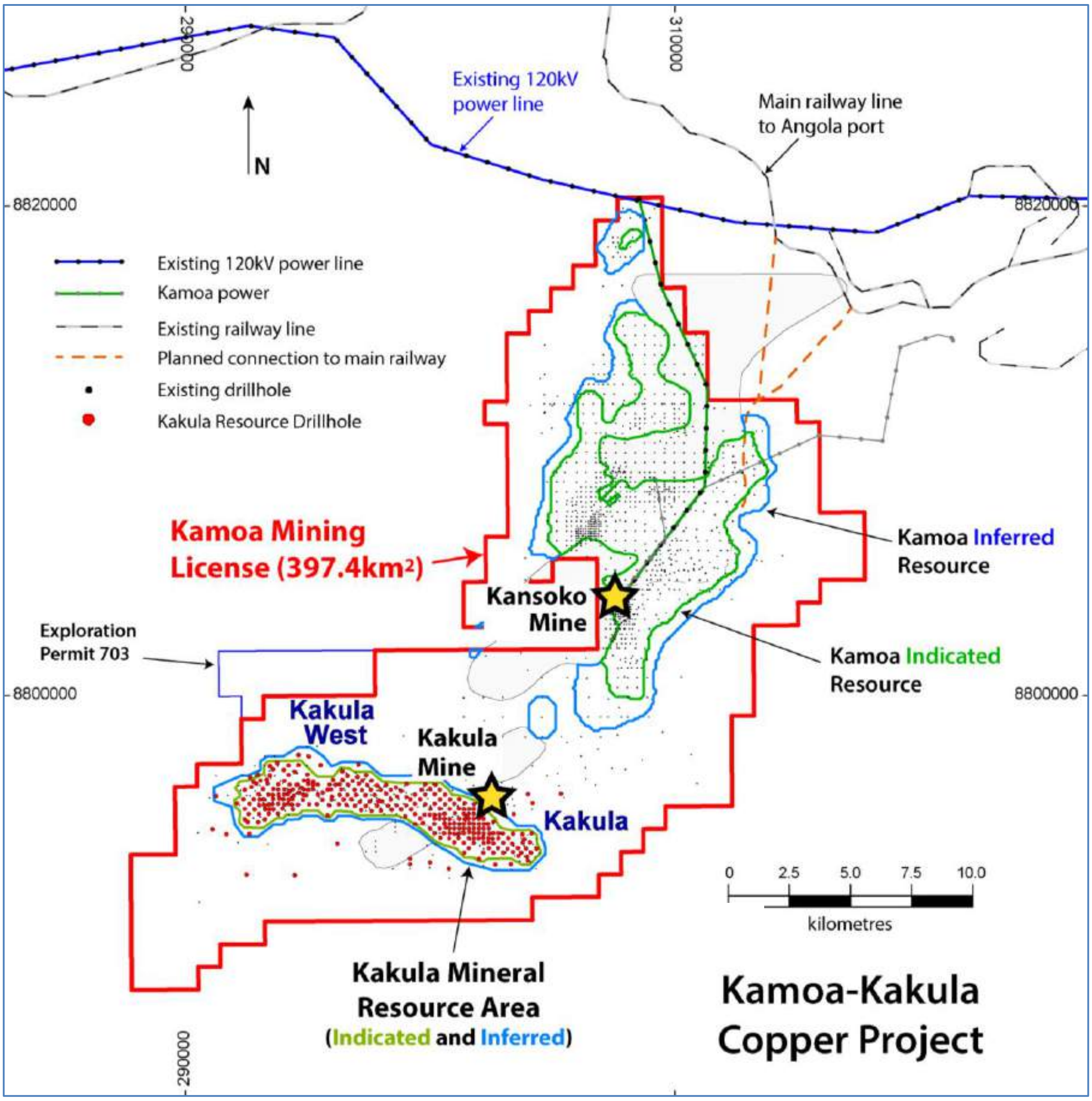


Figure by Kamoa Copper SA, 2019.

At a 1% copper cut-off, the Kakula Mineral Resources cover an area of 27.4 km² with 21.5 km² classified as Indicated and 5.9 km² classified as Inferred. The deposit remains open laterally for expansion that is currently being drilled, and the southern parts of the Kamoa-Kakula exploitation licence area is virtually untested.

The Kakula resource estimations were controlled using four domains based on stratigraphy and two domains based on a combination of stratigraphy and a mineralised envelope. The mineralised envelope was defined using an approximate cut-off grade of 1% TCu, and was divided into two domains based on whether host stratigraphy was siltstone or diamictite.

To account for the undulations of the deposit and ensure that the grade profiles between drill holes align during estimation, drillhole composites and blocks were transformed vertically or “dilated” to a constant thickness that matched the maximum thickness of the mineralisation. This method aligns the top, middle and bottom of the mineralised intervals horizontally for variography and grade estimation using ordinary kriging. This preserves the important vertical grade and mineralogical zonation to allow vertical optimisation during mine design. To adjust for local changes in the trend of the mineralisation laterally, geological controls were used to locally adjust the search orientations during estimation using a Datamine process known as dynamic anisotropy. This was used primarily at Kakula West.

For reporting Mineral Resources, Amec Foster Wheeler used a 1% TCu cut-off grade as a base case. This choice of cut-off is based on many years of experience on the Zambian Copperbelt at mines with similar mineralisation such as Konkola, Nchanga, Nkana, and Mufulira where the 1% cut-off is considered a natural cut-off. The 1% TCu cut-off is also a “natural” cut-off for the Kamoa and Kakula deposits, with most intervals grading a few tenths of a percent copper above and below the SMZ composite and well over 1% Cu within the SMZ composite. To test the 1% cut-off grade for the purposes of assessing reasonable prospects of eventual economic extraction, Amec Foster Wheeler performed a conceptual analysis.

1.12.1 Kamoa-Kakula Mineral Resource Statement

The effective date of the estimate for Kamoa is 27 November 2017, and the cut-off date for drill data is 23 November 2015. The prominent north–north-west-trending West Scarp Fault has a significant vertical offset, and was used to divide the Kakula Mineral Resource model into two sections. The Kakula West deposit is located west of the West Scarp Fault, and Kakula is located east of the West Scarp Fault. The Kakula West model was modelled using drillhole data provided up to 1 November 2018, and Kakula was modelled using drillhole data provided up to 26 January 2018.

Total Mineral Resources for the Kamoa-Kakula Project are summarised in Table 1.2 using a 1.0% TCu cut-off, a minimum vertical height of 3 m, and are reported on a 100% basis. Ivanhoe holds an indirect 39.6% interest in the Project. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Table 1.3 summarises the combined Kakula Mineral Resource (Kakula and Kakula West) at a range of cut-off grades with the base case cut-off of 1.0% TCu highlighted in grey. Mineral Resources in Table 1.3 are not additive to the Mineral Resources in Table 1.2.

Table 1.2 Kamoa and Combined Kakula Indicated and Inferred Mineral Resources

Deposit	Category	Tonnes (millions)	Area (Sq. km)	Copper Grade (%)	Vertical Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
Kamoa	Indicated	759	50.7	2.57	5.5	19,500	43.0
	Inferred	202	19.4	1.85	3.8	3,740	8.2
Kakula	Indicated	628	21.5	2.72	10.5	17,100	37.6
	Inferred	114	5.9	1.59	6.9	1,810	4.0
Total Kamoa-Kakula Project	**Indicated**	**1,387**	**72.2**	**2.64**	**6.9**	**36,600**	**80.6**
	Inferred	**316**	**25.3**	**1.76**	**4.5**	**5,550**	**12.2**

1. Ivanhoe's Mineral Resources Manager, George Gilchrist, Professional Natural Scientist (Pr. Sci. Nat) with the South African Council for Natural Scientific Professions (SACNASP), estimated the Mineral Resources under the supervision of Dr. Harry Parker and Gordon Seibel, both Registered Members (RM) of the Society for Mining, Metallurgy and Exploration (SME), who are the Qualified Persons for the Mineral Resource estimate. The effective date of the estimate for Kamoa is 27 November 2017, and the cut-off date for drill data is 23 November 2015. The Kakula Mineral Resource is a combination of separate Kakula and Kakula West models, with the West Scarp Fault forming the boundary between the two. The effective date of the estimate for Kakula is 13 April 2018, and the cut-off date for the drill data is 26 January 2018. The effective date of the estimate for Kakula West is 10 November 2018, and the cut-off date for the drill data is 1 November 2018. Mineral Resources are reported on a 100% basis (Ivanhoe holds an indirect 39.6% interest in the Project), using the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves. Mineral Resources at Kamoa are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
2. Mineral Resources at Kamoa are reported using a total copper (TCu) cut-off grade of 1% TCu and a minimum vertical thickness of 3 m. There are reasonable prospects for eventual economic extraction under assumptions of a copper price of US$3.00/lb; employment of underground mechanised room-and-pillar and drift-and-fill mining methods; and that copper concentrates will be produced and sold to a smelter. Mining costs are assumed to be US$27/t, and concentrator, tailings treatment, and general and administrative costs (G&A) are assumed to be US$17/t. Metallurgical recovery for Kamoa is estimated to average 84% (86% for hypogene and 81% for supergene). At a 1% TCu cut-off grade, assumed net smelter returns for 100% of Mineral Resource blocks will cover concentrator, tailings treatment, and G&A costs.
3. Mineral Resources at Kakula are reported using a TCu cut-off grade of 1% TCu and an minimum vertical thickness of 3 m. There are reasonable prospects for eventual economic extraction under assumptions of a copper price of US$3.10/lb, employment of underground, mechanised, room-and-pillar and drift-and-fill mining methods, and that copper concentrates will be produced and sold to a smelter. Mining costs are assumed to be US$34/t, and concentrator, tailings treatment, and G&A costs are assumed to be US$20/t. Metallurgical recovery is assumed to average 83% at the average grade of the Mineral Resource. Ivanhoe is studying reducing mining costs using a controlled convergence room-and-pillar method. At a 1% TCu cut-off grade, assumed net smelter returns for 100% of Mineral Resource blocks will cover concentrator, tailings treatment and G&A costs.
4. Reported Mineral Resources contain no allowances for hangingwall or footwall contact boundary loss and dilution. No mining recovery has been applied.
5. Tonnage and contained-copper tonnes are reported in metric units, contained-copper pounds are reported in imperial units, and grades are reported as percentages.
6. Approximate drillhole spacings are 800 m for Inferred Mineral Resources and 400 m for Indicated Mineral Resources.
7. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.

Table 1.3 Sensitivity of the Combined Kakula Mineral Resources to Cut-off Grade (base case at 1% Cu is highlighted)

Indicated Mineral Resource

Cut-off (% Cu)	Tonnage (Mt)	Area (km^2)	Copper Grade (%)	True Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
7.0	39	3.2	8.97	4.1	3,460	7.6
6.0	53	4.4	8.29	4.2	4,390	9.7
5.0	77	6.3	7.41	4.3	5,690	12.6
4.0	107	8.4	6.58	4.5	7,070	15.6
3.0	164	11.7	5.50	4.9	9,010	19.9
2.5	217	14.3	4.82	5.4	10,500	23.1
2.0	319	17.4	3.99	6.5	12,700	28.1
1.5	434	19.4	3.40	8.0	14,700	32.5
1.0	628	21.5	2.72	10.5	17,100	37.6

Inferred Mineral Resource

Cut-off (% Cu)	Tonnage (Mt)	Area (km^2)	Copper Grade (%)	True Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
4.0	1	0.1	4.40	3.4	32	0.1
3.0	5	0.4	3.52	3.9	163	0.4
2.5	11	1.0	3.09	3.7	325	0.7
2.0	23	2.1	2.62	3.9	604	1.3
1.5	47	3.9	2.16	4.3	1,010	2.2
1.0	114	5.9	1.59	6.9	1,810	4.0

1. Ivanhoe's Mineral Resources Manager George Gilchrist, a Fellow of the Geology Society of South Africa and Professional Natural Scientist (Pr. Sci. Nat) with the South African Council for Natural Scientific Professions (SACNASP), estimated the Mineral Resources under the supervision of Dr. Harry Parker and Gordon Seibel, both Registered Members (RM) of the Society for Mining, Metallurgy and Exploration (SME), who are the Qualified Persons for the Mineral Resources. The Kakula Mineral Resource is a combination of separate Kakula and Kakula West resource models, with the West Scarp Fault forming the boundary between the two. The effective date of the estimate for Kakula is 13 April 2018, and the cut-off date for the drill data is 26 January 2018. The effective date of the estimate for Kakula West is 10 November 2018, and the cut-off date for the drill data is 1 November 2018. Mineral Resources are reported using the CIM Definition Standards for Mineral Resources and Reserves (2014), and are reported inclusive of Mineral Reserves on a 100% basis. Ivanhoe holds an indirect 39.6% interest in the Project. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
2. Mineral Resources are reported using a total copper (TCu) cut-off grade of 1% TCu and a minimum vertical thickness of 3 m. There are reasonable prospects for eventual economic extraction under assumptions of a copper price of US$3.10/lb, employment of underground, mechanised, room-and-pillar and drift-and-fill mining methods, and that copper concentrates will be produced and sold to a smelter. Mining costs are assumed to be US$34/t. Concentrator, tailings treatment and general and administrative (G&A) costs are assumed to be US$20/t. Metallurgical recovery is assumed to average 83%. Ivanhoe is studying reducing mining costs using a controlled convergence room-and-pillar method. At a 1% TCu cut-off grade, 100% of the assumed net smelter returns for Mineral Resource blocks will cover concentrator, tailings treatment and G&A costs.

3. Reported Mineral Resources contain no allowances for hangingwall or footwall contact boundary loss and dilution. No mining recovery has been applied.
4. Tonnage and contained-copper tonnes are reported in metric units, contained-copper pounds are reported in imperial units, and grades are reported as percentages.
5. Rounding as required by reporting guidelines may result in apparent differences between tonnes, grade and contained metal content.

1.12.2 Factors Which May Affect the Resource Estimates

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

- Drill spacing.
  - The drill spacing at the Kamoa and Kakula deposits is insufficient to determine the effects of local faulting on lithology and grade continuity assumptions. Local faulting could disrupt the productivity of a highly-mechanised operation. In addition, the amount of contact dilution related to local undulations in the SMZ has yet to be determined for both deposits. Ivanhoe plans to study these risks with the declines currently in progress at Kamoa and Kakula.
  - Delineation drill programs at the Kamoa deposit will have to use a tight (approximately 50 m) spacing to define the boundaries of mosaic pieces (areas of similar stratigraphic position of SMZs) in order that mine planning can identify and deal with these discontinuities. At the Kakula deposit, the mineralisation appears more continuous compared to Kamoa.
- Assumptions used to generate the data for consideration of reasonable prospects of eventual economic extraction for the Kamoa deposit.
  - Mining recovery could be lower and dilution increased where the dip locally increases on the flanks of the domes. The exploration decline should provide an appropriate trial of the conceptual room-and-pillar mining method on the Kamoa deposit in terms of costs, dilution, and mining recovery. The decline will also provide access to data and metallurgical samples at a bulk scale that cannot be collected at the scale of a drill sample.
- Assumptions used to generate the data for consideration of reasonable prospects of eventual economic extraction for the Kakula deposit.
  - A controlled convergence room-and-pillar technique is being studied which provides the opportunity for reduced costs.
- Metallurgical recovery assumptions at Kamoa.
  - Metallurgical testwork at the Kamoa deposit indicates the need for multiple grinding and flotation steps. Variability testwork has been conducted on only portions of the Kamoa deposit. Additional variability testing is needed to build models relating copper mineralogy to concentrate grade and improve the recovery modelling especially for supergene mineralisation.

- Metallurgical recovery assumptions at Kakula.
  - Metallurgical testwork at the Kakula deposit indicates the need for fine mainstream grinding and additional fine grinding of concentrate. Low entrainment cleaning proved to be beneficial to the flow sheet in terms of grade and silica control.
  - A recovery model has been developed based on correlations obtained from preliminary variability testwork to determine the relationship between mass pull and concentrate upgrade ratio.
  - Preliminary testwork on the Kakula West material indicated that the material can be successfully upgraded using the Kakula 2019 PFS flow sheet.
  - Further testwork on the Kamoa material indicated that the Kansoko and Kakula material can be processed in a common concentrator circuit.
- Exploitation of the Kamoa-Kakula Project requires building a greenfields project with attendant 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 Kamoa and Kakula Mineral Resource estimates.
- Commodity prices and exchange rates.
- Cut-off grades.

1.13 Targets for Further Exploration

A target for further exploration (referred to as an exploration target for the purposes of this Report) has been identified adjacent to the Kamoa Mineral Resource. This target is referred to as the Kamoa-Makalu exploration target.

The area inside the model perimeter surrounding the Indicated and Inferred Mineral Resources at Kamoa is considered an exploration target. The ranges of the Kamoa-Makalu exploration target tonnages and grades are summarised in Table 1.4. Tonnage and grade ranges were estimated using an inverse distance weighting and applying a +/-20% variance to the resulting tonnage and grade estimate.

Amec Foster Wheeler cautions that the potential quantity and grade of the exploration target is conceptual in nature, and that it is uncertain if additional drilling will result in the exploration target being delineated as a Mineral Resource. No exploration targets have been defined for Kakula at this time.

Table 1.4 Tonnage and Grade Ranges for Kamoa-Makalu Exploration Target

Target	Low-range Tonnage (Mt)	High-range Tonnage (Mt)	Low-range Grade (% Cu)	High-range Grade (% Cu)
Kamoa	480	720	1.5	2.3

1.14 Mineral Reserves

1.14.1 Kamoa-Kakula Project Mineral Reserve

The Kamoa-Kakula Project Mineral Reserve includes the ore from both the Kakula Mine on the Kakula Deposit and Kansoko Mine at the Kamoa Deposit. The tonnes and grades were calculated for the mining blocks, and allowances for unplanned dilution and mining recovery were applied to calculate the Mineral Reserve Statement. The Total Probable Mineral Reserves are summarised in Table 1.5.

Table 1.5 Kamoa-Kakula Project Mineral 2019 Mineral Reserve

Classification	Ore (Mt)	Copper (%)	Copper (Contained Mlb)	Copper (Contained kt)
Proven Mineral Reserve	–	–	–	–
Probable Mineral Reserve	245.0	4.63	25,000	11,340
Mineral Reserve	245.0	4.63	25,000	11,340

1. Effective date of the all Mineral Reserves is 1 February 2019.
2. Mineral Reserves are the total for the Kakula and Kansoko Mines.
3. The copper price used for calculating the financial analysis is long-term copper at US$3.10/lb. The analysis has been calculated with assumptions for smelter refining and treatment charges, deductions and payment terms, concentrate transport, metallurgical recoveries, and royalties.
4. For mine planning, the copper price used to calculate block model net smelter returns (NSRs) was US$3.00/lb.
5. An elevated cut-off of US$100.00/t NSR was used to define the stoping panels. A marginal cut-off of US$80.00/t NSR was used to define ore and waste.
6. Indicated Mineral Resources were used to estimate Probable Mineral Reserves.
7. Tonnage and grade estimates include dilution and recovery allowances.
8. The Mineral Reserves reported above are not additive to the Mineral Resources.

1.14.2 Kakula 2019 PFS Mineral Reserve

The Kakula 2019 PFS Mineral Reserve targeted in the mine plan focused on maximizing the grade profile for a 6 Mtpa full production rate for 15 years, a 5-year ramp-up, plus an 85% extraction and recovery allowance. As such, a range of net smelter return (NSR) cut-offs were evaluated that identified a targeted resource of approximately 125 Mt at the highest NSR.

Tonnes and grades were calculated for the mining blocks, and allowances for unplanned dilution and mining recovery were applied to calculate the Mineral Reserve. The Kakula 2019 PFS Probable Mineral Reserves are shown in Table 1.6.

Table 1.6 Kakula 2019 PFS Mineral Reserve

Classification	Ore (Mt)	Copper (%)	Copper (Contained Mlb)	Copper (Contained kt)
Proven Mineral Reserve	–	–	–	–
Probable Mineral Reserve	119.7	5.48	14,475	6,566
Mineral Reserve	119.7	5.48	14,475	6,566

1. Effective date of the Kakula Mineral Reserves is 1 February 2019.
2. The copper price used for calculating the financial analysis is long-term copper at US$3.10/lb. The analysis has been calculated with assumptions for smelter refining and treatment charges, deductions and payment terms, concentrate transport, metallurgical recoveries, and royalties.
3. For mine planning, the copper price used to calculate block model NSRs was US$3.00/lb.
4. An elevated cut-off of US$100.00/t NSR was used to define the stoping panels. A marginal cut-off of US$80.00/t NSR was used to define ore and waste.
5. Indicated Mineral Resources were used to estimate Probable Mineral Reserves.
6. Tonnage and grade estimates include dilution and recovery allowances.
7. The Mineral Reserves reported above are not additive to the Mineral Resources.

1.14.3 Kamoa 2019 PFS Mineral Reserve

The Mineral Reserve estimate in the Kamoa 2019 PFS 6 Mtpa scenario is based on the resource block model developed by Ivanhoe under the direction of AMEC and provided to Stantec in July 2016 (file name: kam14a160309). NSR values were calculated and inserted into the model by Ivanplats and OreWin Pty Ltd (OreWin). Only the Indicated portion of the resource was used in estimating the Mineral Reserve. None of the resources are currently classified as Measured.

The reserve focused on maximizing the grade profile for a 6 Mtpa (total rock) production rate for approximately 22 years. As such, a range of NSR cut-offs were evaluated to develop the reserve statement to get approximately 125.2 Mt at the highest NSR. This strategy provides opportunities for either a longer mine life or ramping up to higher production rates to utilise more of the resource. The final LOM schedule resulted in 17 years of full production and a 26-year LOM.

Tonnes and grades were calculated for panels, and allowances for unplanned dilution and mining recovery were applied to calculate the Probable Ore Reserves. The total Mineral Reserves for the Kamoa 2019 PFS are summarised in Table 1.7.

Table 1.7 Kamoa 2019 PFS Mineral Reserve

Classification	Tonnage (Mt)	Copper (%)	Contained Copper in Ore (Mlb)	Contained Copper in Ore (kt)
Proven Mineral Reserve	–	–	–	–
Probable Mineral Reserve	125.2	3.81	10,525	4,774
Mineral Reserve	**125.2**	**3.81**	**10,525**	**4,774**

1. Effective date of the Kamoa Mineral Reserve is 1 February 2019.
2. The copper price used for calculating the financial analysis is long-term copper at US$3.10/lb. The analysis has been calculated with assumptions for smelter refining and treatment charges, deductions and payment terms, concentrate transport, metallurgical recoveries and royalties.
3. For mine planning, the copper price used to calculate block model net smelter returns (NSRs) was US$3.00/lb.
4. An elevated cut-off of US$100.00/t NSR was used to define the stoping panels. A cut-off of US$80.00/t NSR was used to define ore and waste for the mine plan.
5. Indicated Mineral Resources were used to estimate Probable Mineral Reserves.
6. Tonnage and grade estimates include dilution and recovery allowances.
7. The Mineral Reserves reported above are not additive to the Mineral Resources.

1.15 Kakula 2019 PFS

Ivanhoe is developing twin declines at the Kakula Mine. Once in production, one will be a service decline for the transport of personnel and materials into the mine, and the second will be a conveyor decline for rock handling and transport of personnel and materials out of the mine. The base case described in the Kakula 2019 PFS is the construction and operation of an underground mine, concentrator processing facilities, and associated infrastructure. The Kakula 2019 PFS production is planned to be an average of 6 Mtpa ore over a production period of 25 years.

1.15.1 Kakula 2019 PFS Mining

The Kakula 2019 PFS mine design access is via a pair of declines on the north side and a single decline on the south side of the deposit. One of the north declines will serve as the primary mine access while the other will include the conveyor haulage system. The conveyor decline has planned dimensions of 7 m wide x 6 m high, and the service decline has planned dimensions of 5.5 m wide x 6 m high. The south decline will serve as a secondary operational ingress/egress and will facilitate critical early mine development. The south decline has planned dimensions of 5.5 m wide x 6 m high. From the bottom of the north and south declines, a pair of perimeter drifts will be driven to the east and west extremities of the deposit and will serve as the primary accesses to the production areas. These drifts will also be used as the primary intake and exhaust ventilation circuits and will connect with a series of intake and exhaust ventilation shafts. The primary ore handling system will include perimeter conveyor drifts and load-out points along the north side of the deposit. The perimeter conveyor drifts will terminate at the main conveyor decline. Connection drifts between the north and south perimeter drifts will provide access and ventilation to the planned mining areas. Mine access and primary development are shown in Figure 1.6. The mining methods for the Kakula deposit are drift-and-fill using paste backfill and room-and-pillar. Approximately 99% of the deposit will be mined using drift-and-fill. Table 1.8 shows the Kakula 2019 PFS Mineral Reserves by mining method.

Table 1.8 Kakula Mineral Reserves by Mining Method

Ore Source	Ore (Mt)	Development (km)	NSR ($/t)	Cu (%)
Ore Development	3.9	37	201.43	4.66
Drift-and-Fill	113.9	1,486	241.00	5.54
Room-and-Pillar	2.0	36	163.41	3.81
Total Ore	119.7	1,559	238.44	5.48

Rounding may result in apparent differences between tonnes and, grade.

Figure 1.6 Kakula 2019 PFS Mine Development

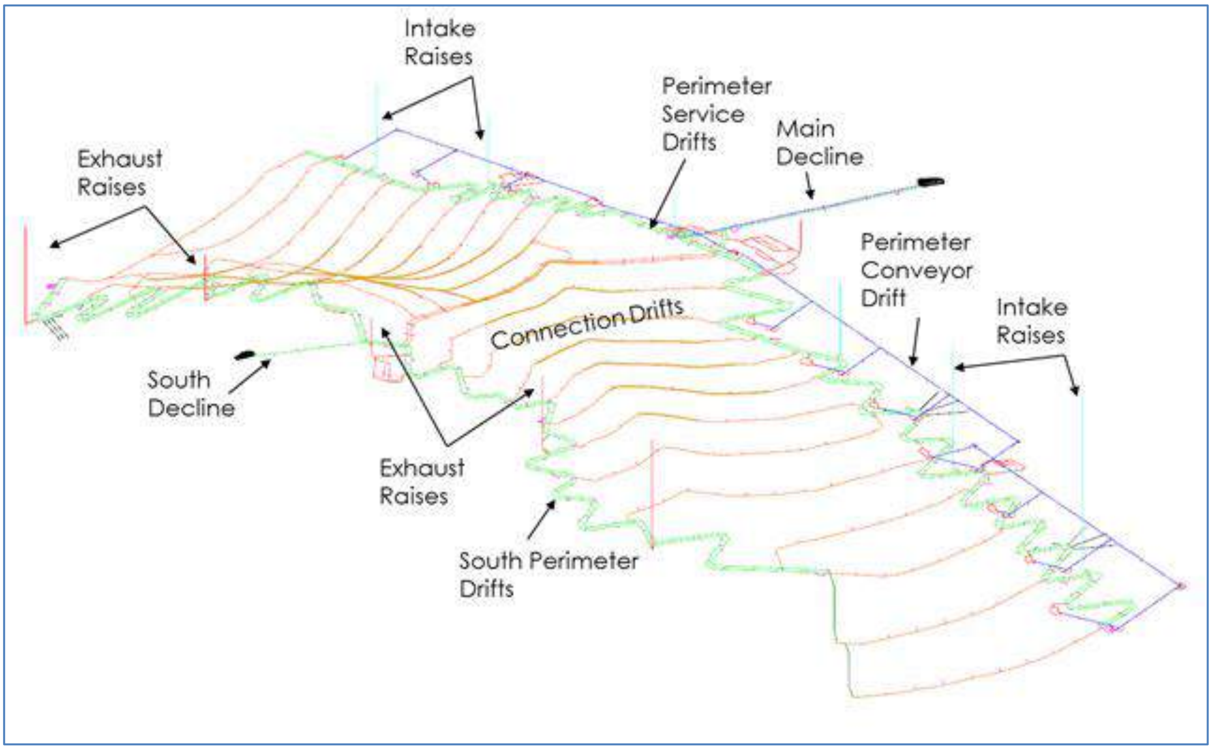


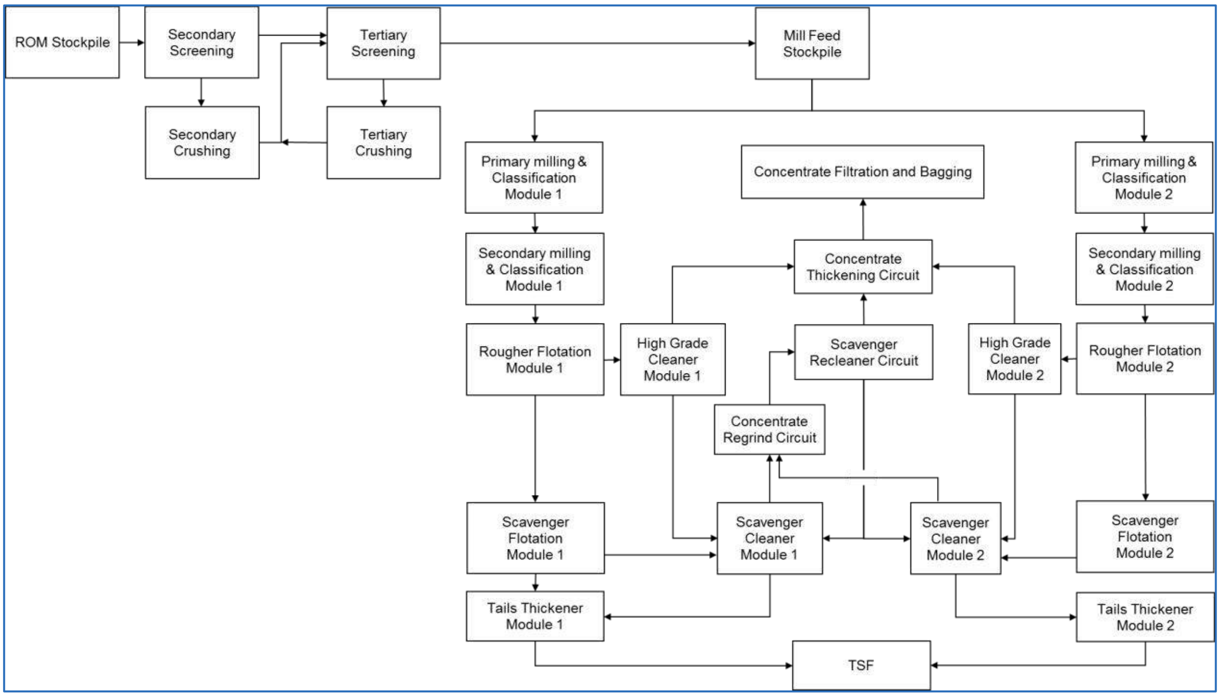
Figure by Stantec, 2019.

1.15.2 Kakula 2019 PFS Process

The Kakula concentrator would be constructed in a phased approach with two three Mtpa modules as the mining operations ramp-up to full production of 6 Mtpa.

The flow sheet for the Kakula concentrator is shown in Figure 1.7. The Kakula concentrator design incorporates a run-of-mine stockpile, followed by two stages of screening and crushing on surface. The crushed material with a size distribution of 80% passing (or P_{80}) eight millimetres (mm), is fed into a two-stage ball-milling circuit for further size reduction to a target grind size 80% passing 53 micrometres (µm). The milled slurry will be passed through a rougher and scavenger flotation. The high-grade, or fast-floating rougher concentrate, and medium-grade, or slow-floating scavenger concentrate, will be collected separately. The rougher concentrate is upgraded in the low entrainment high-grade cleaner stage to produce a high-grade concentrate. The medium-grade scavenger concentrate together with the tailings from the rougher cleaning stage and the tailings from the scavenger recleaner stage will be combined and further upgraded in the scavenger cleaner circuit. The concentrate produced from the scavenger cleaner circuit, representing roughly 12% of the mill feed, will be re-ground to a P_{80} of 10 µm prior to final cleaning in the low entrainment scavenger recleaner stage. The scavenger recleaner concentrate then will be combined with the high-grade cleaner concentrate to form the final concentrate. The final concentrate will be thickened before being pumped to the concentrate filter where the filter cake will then be bagged for shipment to market.

Figure 1.7 Kakula Process Flow Sheet



1.15.3 Kakula 2019 PFS Transport

A phased logistics solution is proposed in the Kakula 2019 PFS. Initially the corridor between southern DRC and Durban in South Africa is viewed as the most attractive and reliable export route.

In the Kamoa-Kakula 2019 PEA it is assumed that rail is rehabilitated between Kolwezi and Dilolo, a town near the DRC Angolan border and then , production from the Kamoa-Kakula Project is transported by rail to the port of Lobito in Angola..

1.15.4 Kakula 2019 PFS Results

The Kakula 2019 PFS describes the initial phase of the Kakula development and presents the first Mineral Reserve for the Kakula Deposit. The Kakula 2019 PFS evaluate the development of a 6 Mtpa underground mine and surface processing complex at the Kakula Deposit. The development scenario of the Kakula Mine on the Kakula Deposit is shown in Figure 1.8.

The Kakula 2019 PFS has an average annual production rate of 291,000 tonnes of copper at a mine site cash cost of US$0.46/lb copper and total cash cost of US$1.11/lb copper for the first ten years of operations. The preproduction capital cost of US$1.1 billion for this option would result in an after-tax net present value at an 8% discount rate (NPV8%) of US$5.4 billion.

Figure 1.8 Kakula 2019 PFS 6 Mtpa Development Scenario

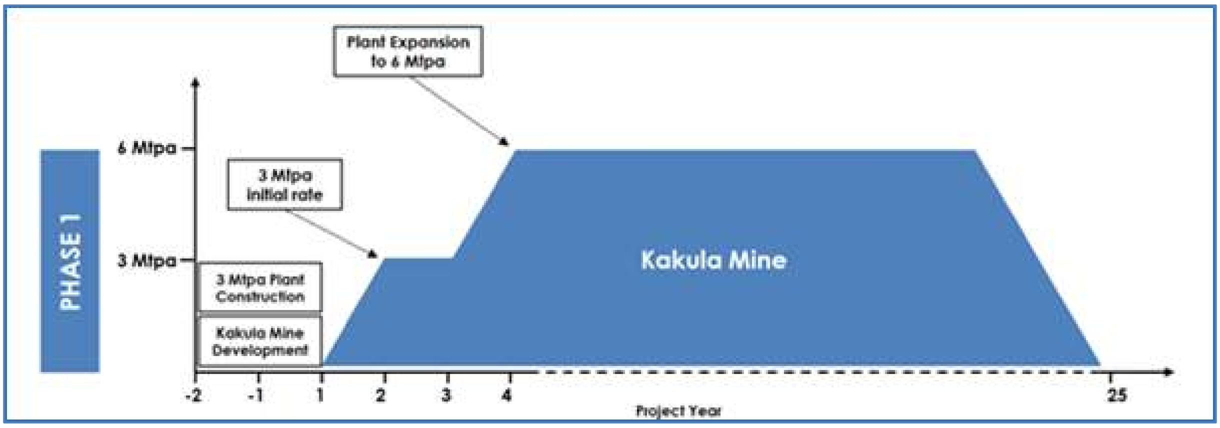


Figure by OreWin, 2019.

A summary of the key results for the Kakula 2019 PFS scenario are:

- Very-high-grade initial phase of production is projected to have a grade of 7.1% copper in Year 2 and an average grade of 6.4% copper over the initial 10 years of operations, resulting in estimated average annual copper production of 224,000 tonnes.
- Initial capital cost, including contingency, is estimated at US$1.1 billion.
- Average total cash cost of US$1.11/lb of copper during the first 10 years.
- After-tax NPV, at an 8% discount rate, of US$5.4 billion.
- After-tax internal rate of return (IRR) of 46.93%, and a payback period of 2.6 years.
- Kakula is expected to produce a very-high-grade copper concentrate in excess of 50% copper, with extremely low arsenic levels.

For the Kakula 2019 PFS the Kakula mill would be constructed in two smaller phases of 3 Mtpa each as the mining operations ramp-up to full production of 6 Mtpa. The life-of-mine (LOM) production scenario provides for 119.7 Mt to be mined at an average grade of 5.48% copper, producing 9.8 Mt of high-grade copper concentrate, containing approximately 12.3 billion pounds of copper. The economic analysis uses a long-term price assumption of US$3.10/lb of copper and returns an after-tax NPV at an 8% discount rate of US$5.4 billion. It has an after-tax IRR of 46.93% and a payback period of 2.6 years. The estimated initial capital cost, including contingency, is US$1.1 billion. The capital expenditure for off-site power, which is included in the initial capital cost, includes a US$64 M advance payment to the DRC state-owned electricity company, SNEL, to upgrade two hydropower plants (Koni and Mwadingusha) to provide the Kamoa-Kakula Project with access to clean electricity for its planned operations. Mwadingusha is being upgraded first. The work is being led by Stucky Ltd., of Switzerland; the advance payment will be recovered through a reduction in the power tariff. Key results of the Kakula 2019 PFS for a single 6 Mtpa mine are summarised in Table 1.9. Table 1.10 summarises the financial results. The mining production statistics are shown in Table 1.11. The Kakula 2019 PFS 6 Mtpa mill feed and copper grade profile for the LOM are shown in Figure 1.9 and the concentrate and metal production for the LOM are shown in Figure 1.10.

Table 1.9 Kakula 2019 PFS Results Summary

Item	Unit	Total
Total Processed		
Quantity Milled	kt	119,728
Copper Feed Grade	%	5.48
Total Concentrate Produced		
Copper Concentrate Produced	kt (dry)	9,776
Copper Concentrate	kt (dry)	9,776
Copper Recovery	%	85.35
Copper Concentrate Grade	%	57.32
Contained Copper in Concentrate	Mlb	12,354
Contained Copper in Concentrate	kt	5,604
Peak Annual Recovered Copper Production	kt	360
Ten Year Average		
Copper Concentrate Produced	kt (dry)	508
Contained Copper in Concentrate	kt	291
Mine-Site Cash Cost	US$/lb Payable Cu	0.46
Total Cash Cost	US$/lb Payable Cu	1.11
Key Financial Results		
Peak Funding	US$M	1,099
Initial Capital Costs	US$M	1,078
Expansion Capital Costs	US$M	778
Sustaining Capital Cost	US$M	1,295
Mine Site Cash Cost	US$/lb Payable Cu	0.59
Total Cash Costs After Credits	US$/lb Payable Cu	1.24
Site Operating Costs	US$/t Milled	59.44
After-Tax NPV8%	US$M	5,440
After-Tax IRR	%	46.9
Project Payback Period	Years	2.6
Initial Project Life	Years	25

Table 1.10 Kakula 2019 PFS Financial Results

	Discount Rate	Before Taxation	After Taxation
Net Present Value (US$M)	Undiscounted	19,317	13,575
	4.0%	12,053	8,411
	6.0%	9,693	6,733
	8.0%	**7,875**	**5,440**
	10.0%	6,457	4,432
	12.0%	5,336	3,635
Internal Rate of Return	–	55.6%	46.9%
Project Payback Period (Years)	–	2.4	2.6

Table 1.11 Kakula 2019 PFS Production and Processing

Item	Unit	Total LOM	Years 1–5	Years 1–10	LOM Average
Total Processed					
Quantity Milled	kt	119,728	4,589	5,321	4,789
Copper Feed Grade	%	5.48	6.79	6.39	5.48
Total Concentrate Produced					
Copper Concentrate Produced	kt (dry)	9,776	465	508	391
Copper Concentrate	kt (dry)	9,776	465	508	391
Copper Recovery	%	85.35	85.58	85.64	85.35
Copper Concentrate Grade	%	57.32	57.32	57.32	57.32
Contained Copper in Concentrate					
Copper	Mlb	12,354	588	642	494
Copper	kt	5,604	267	291	224
Payable Copper in Concentrate					
Copper	Mlb	12,074	575	628	483
Copper	kt	5,477	261	285	219
Payable Copper					
Copper	Mlb	12,074	575	628	483
Copper	kt	5,477	261	285	219

Figure 1.9 Kakula 2019 PFS Process Production

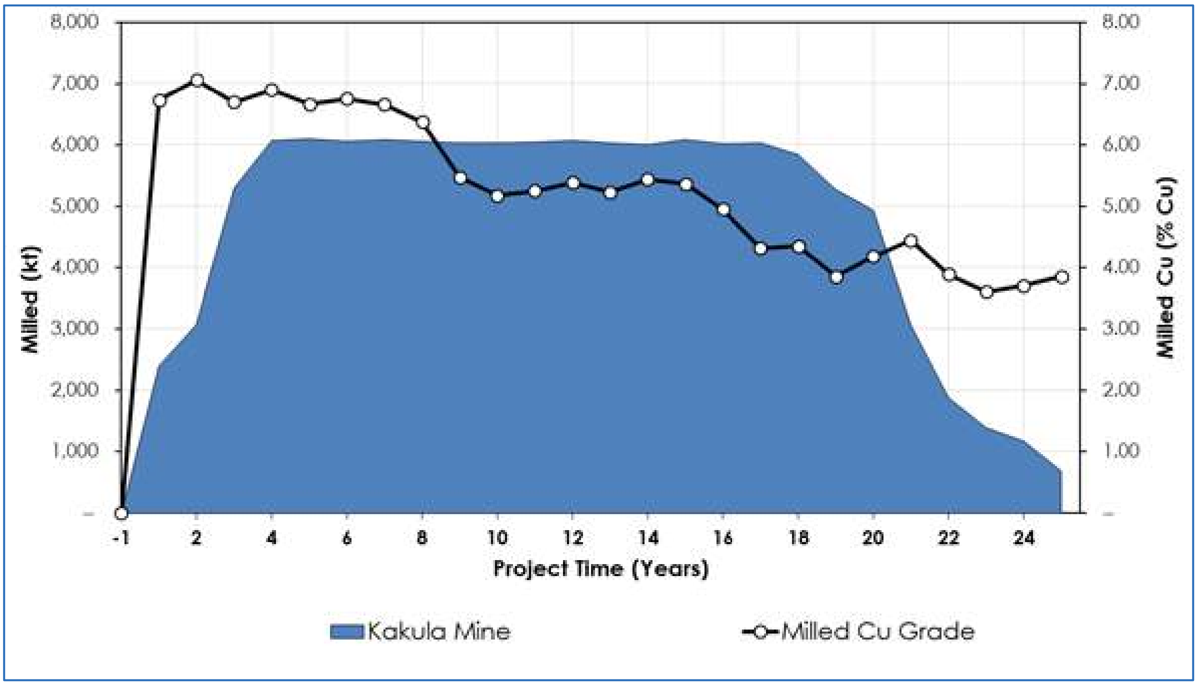


Figure by OreWin, 2019.

Figure 1.10 Kakula 2019 PFS Concentrate and Metal Production

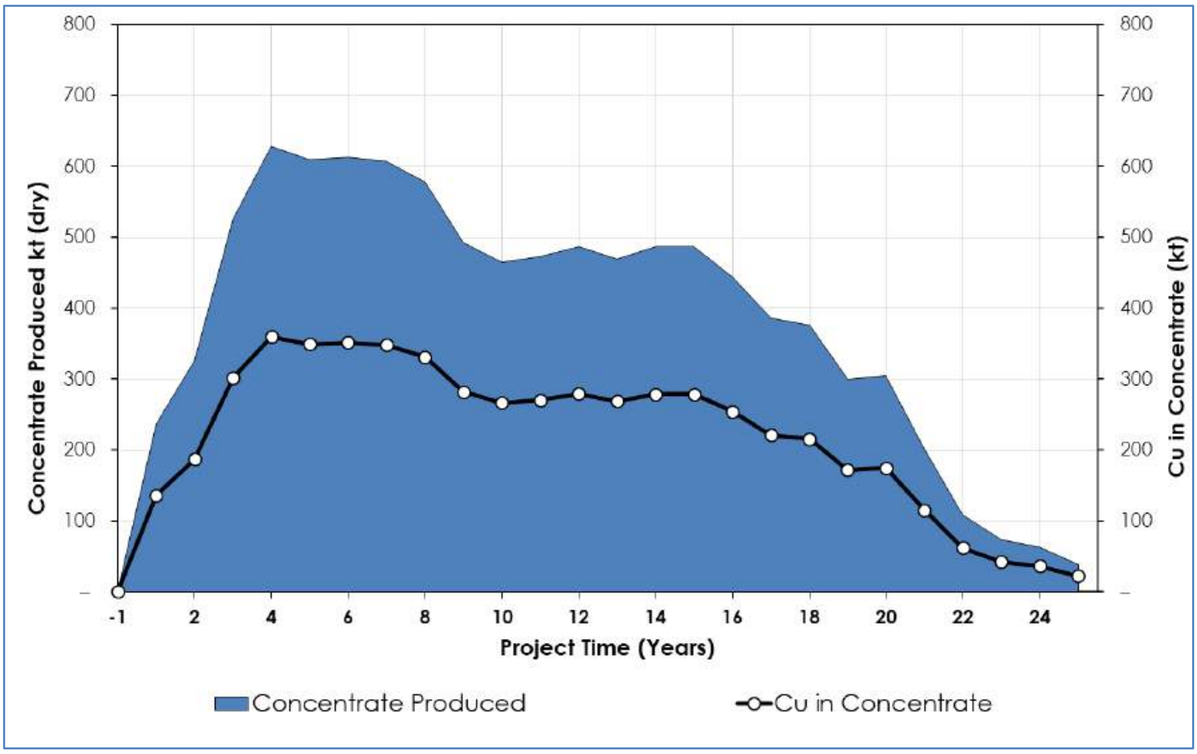


Figure by OreWin, 2019.

The annual and cumulative cash flows are shown in Figure 1.11 (annual cash flow is shown on the left vertical axis and cumulative cash flow on the right axis).

Figure 1.11 Kakula Mine Projected Cumulative Cash Flow

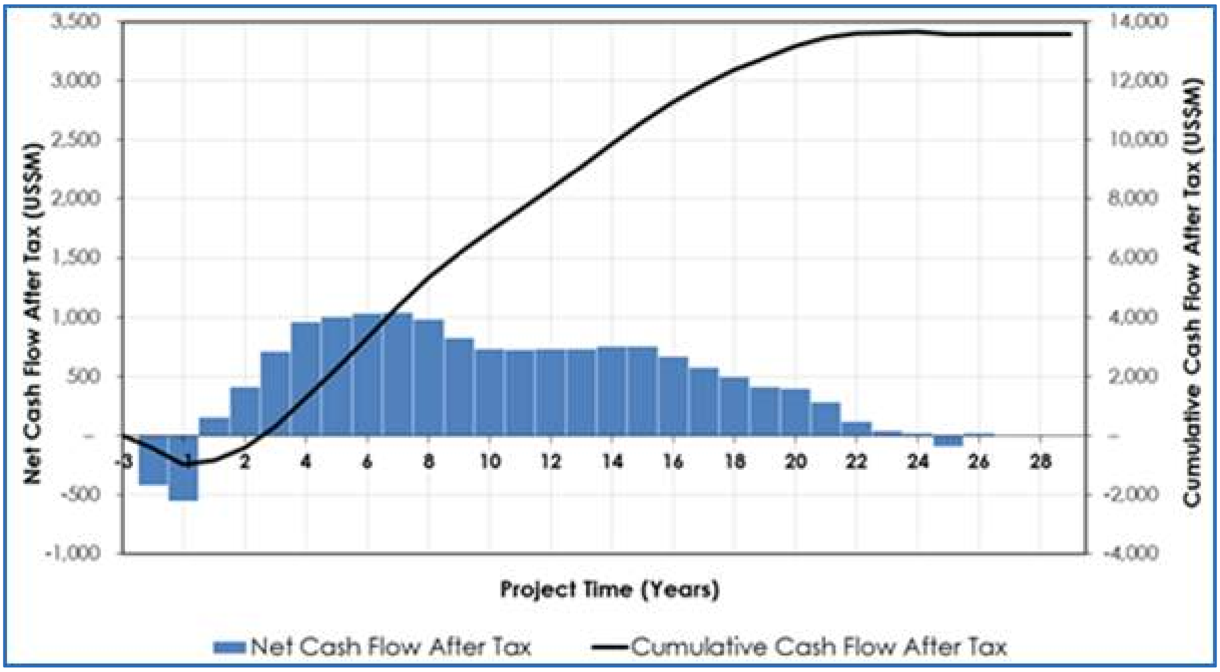


Figure by OreWin, 2019.

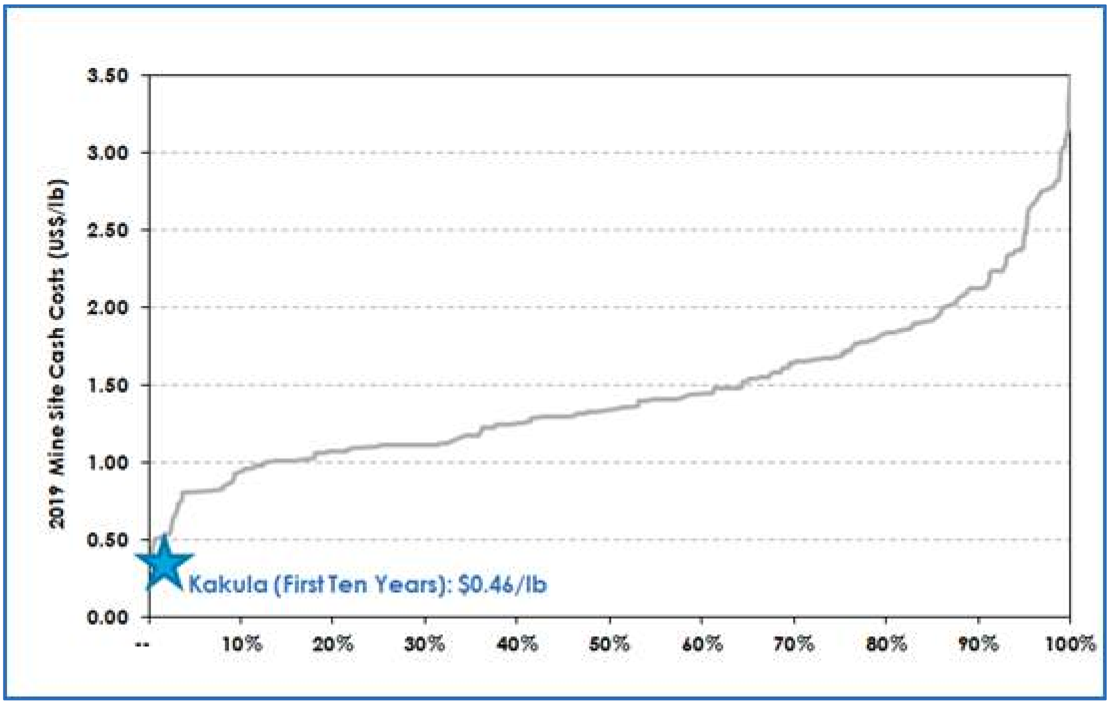
Figure 1.12 shows the average mine-site cash cost during the first 10 years of the Kakula 2019 PFS on Wood Mackenzie's industry cost curve. This figure represents mine-site cash costs that reflect the direct cash costs of producing paid concentrate or cathode incorporating mining, processing and mine-site G&A costs.

Figure 1.13 shows the C1 pro-rata copper cash costs of the Kakula 2019 PFS on Wood Mackenzie's industry cost curve. This figure represents C1 pro-rata cash costs that reflect the direct cash costs of producing paid copper incorporating mining, processing, mine-site G&A and offsite realization costs, having made appropriate allowance for the costs associated with the co-product revenue streams.

For both charts, the Kamoa Kakula IDP19 was not reviewed by Wood Mackenzie prior to filing, and information was sourced the data from public disclosures.

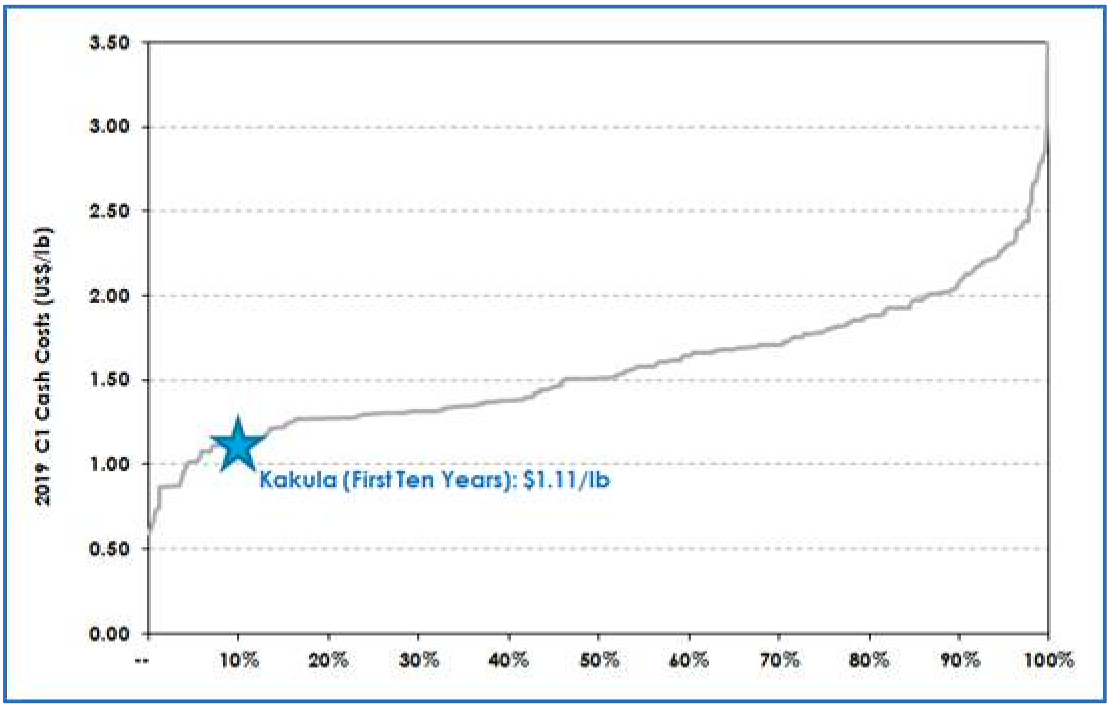
Table 1.12 summarises unit operating costs. Table 1.13 provides a breakdown of revenue and operating costs. Capital costs for the project are detailed in Table 1.14.

Figure 1.12 2019 Mine-Site Cash Costs (Includes All Operational Costs at Mine Site)



Source: Wood Mackenzie 2019

Figure 1.13 2019 C1 Copper Cash Costs



Source: Wood Mackenzie 2019.

Table 1.12 Kakula 2019 PFS Unit Operating Costs

	Payable Cu (US$/lb)		
	Years 1–5	Years 1–10	LOM Average
Mine Site	0.43	0.46	0.59
Transport	0.31	0.31	0.31
Treatment & Refining Charges	0.15	0.15	0.15
Royalties & Export Tax	0.20	0.20	0.20
Total Cash Costs	**1.08**	**1.11**	**1.24**

Table 1.13 Kakula 2019 PFS Revenue and Operating Costs

	Total LOM	Years 1–5	Years 1–10	LOM Average
	US$M	US$/t Milled		
Revenue				
Copper in Concentrate	37,429	388.32	365.60	312.62
Gross Sales Revenue	**37,429**	**388.32**	**365.60**	**312.62**
Less: Realisation Costs				
Transport	3,707	38.46	36.21	30.96
Treatment and Refining	1,770	18.37	17.29	14.79
Royalties and Export Tax	2,403	24.93	23.47	20.07
Total Realisation Costs	**7,880**	**81.76**	**76.97**	**65.82**
Net Sales Revenue	**29,549**	**306.57**	**288.63**	**246.80**
Site Operating Costs				
Underground Mining	4,585	34.37	35.19	38.30
Processing	1,549	12.86	12.35	12.94
Tailings	25	0.20	0.18	0.21
General and Administration	816	6.38	6.02	6.82
SNEL Discount	-212	-2.45	-2.47	-1.72
Customs Duties	347	2.68	2.70	2.90
Total	**7,111**	**54.04**	**53.97**	**59.44**
Net Operating Margin	**22,438**	**252.53**	**234.66**	**187.36**
Net Operating Margin	**75.94%**	**82.37%**	**81.30%**	**75.92%**

Table 1.14 Kakula 2019 PFS Capital Costs

Description	**Initial Capital US$M**	**Expansion Capital US$M**	**Sustaining Capital US$M**	**Total US$M**
Mining				
Underground Mining	287	339	633	1,259
Capitalised Preproduction	107	–	–	107
Subtotal	**394**	**339**	**633**	**1,367**
Power				
Power Supply Off Site	64	–	–	64
Subtotal	**64**	**–**	**–**	**64**
Concentrate and Tailings				
Process Plant	190	125	219	534
Tailings	24	15	83	122
Subtotal	**214**	**140**	**303**	**656**
Infrastructure				
Plant Infrastructure	109	124	187	419
Subtotal	**109**	**124**	**187**	**419**
Indirects				
EPCM	56	40	4	100
Owners Cost	103	25	–	128
Customs Duties	29	22	38	90
Closure	–	–	69	69
Subtotal	**188**	**88**	**111**	**387**
Capital Expenditure Before Contingency	**968**	**690**	**1,234**	**2,893**
Contingency	110	88	62	259
Capital Expenditure After Contingency	**1,078**	**778**	**1,295**	**3,152**

Totals have been rounded.

Figure 1.14 compares the capital intensity for large-scale copper projects. The figure shows projects identified by Wood Mackenzie as recently approved, probable or possible projects reported with nominal copper production capacity in excess of 200 ktpa (based on public disclosure and information gathered in the process of routine research by Wood Mackenzie). The estimates are based on public disclosure and information gathered by Wood Mackenzie. The Kamoa Kakula IDP19 was not reviewed by Wood Mackenzie prior to filing.

Figure 1.14 Capital Intensity for Large-Scale Copper Projects

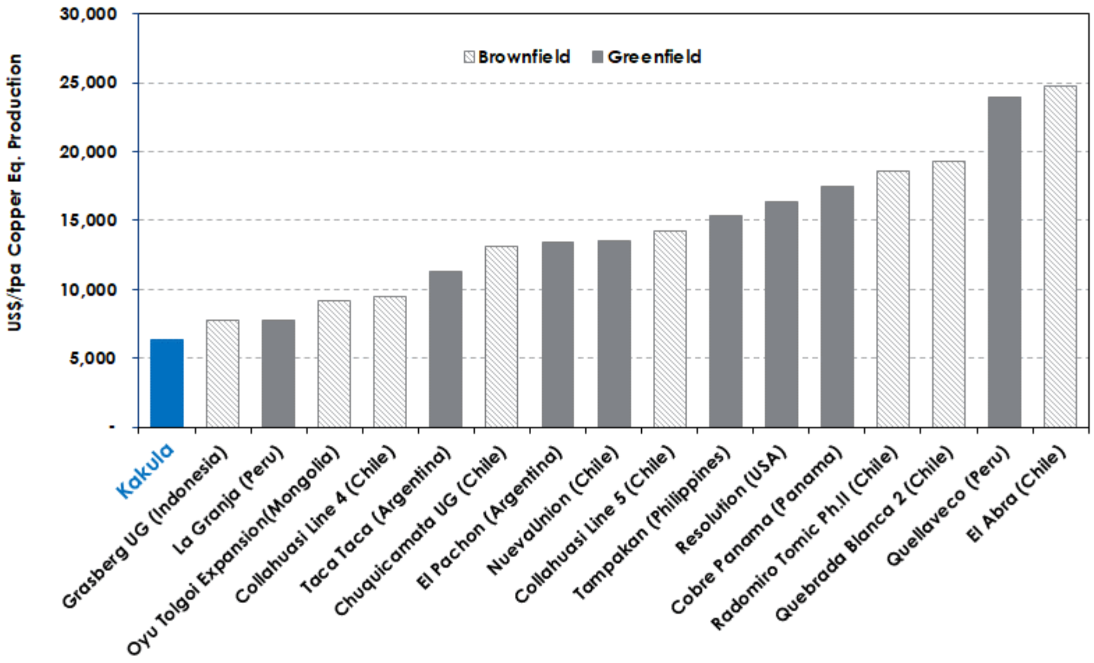


Figure by Ivanhoe, 2019. Source: Wood Mackenzie.

1.16 Kamoa 2019 PFS

The Kamoa 2019 PFS is an update of the work previously called the Kamoa 2017 PFS. The mine design, production, process plant and infrastructure have remained the same. Costs were updated by applying a 2% escalation to the total costs and the copper price was revised based on current long-term forecasts. Ivanhoe has developed twin declines at the Kansoko Mine on the Kansoko areas of the Kamoa deposit. Once in production, one will be a service decline for the transport of personnel and materials into the mine, and the second will be a conveyor decline for rock handling and transport of personnel and materials out of the mine. The Kansoko Mine on the Kamoa Deposit has a Mineral Reserve that was previously stated in the Kamoa 2017 PFS and is updated in the Kamoa 2019 PFS. The Kamoa 2019 PFS is the planning for the development and operation of the Kansoko zones at Kamoa.

The base case described in the Kamoa 2019 PFS is the construction and operation of an underground mine, concentrator processing facilities, and associated infrastructure. The Kamoa 2019 PFS production is planned to be an average of 6 Mtpa ore over a production period of 26 years.

1.16.1 Kamoa 2019 PFS Mining

A probable Mineral Reserve of approximately 125.2 Mt grading at 3.81% Cu has been defined in multiple mining zones to support a 6 Mtpa rate over a 26-year mine life. These ore zones occur at depths ranging from approximately 60 m to 1,235 m. Access to the mine will be via twin declines. Main declines and ventilation raises are shown in Figure 1.15. Mining will be performed using the room-and-pillar mining method in the mineralised zone between 60–150 m and controlled convergence room-and-pillar for mineralised zones below 150 m.

Figure 1.15 Kamoa 2019 PFS Kansoko Mine Access and Ventilation

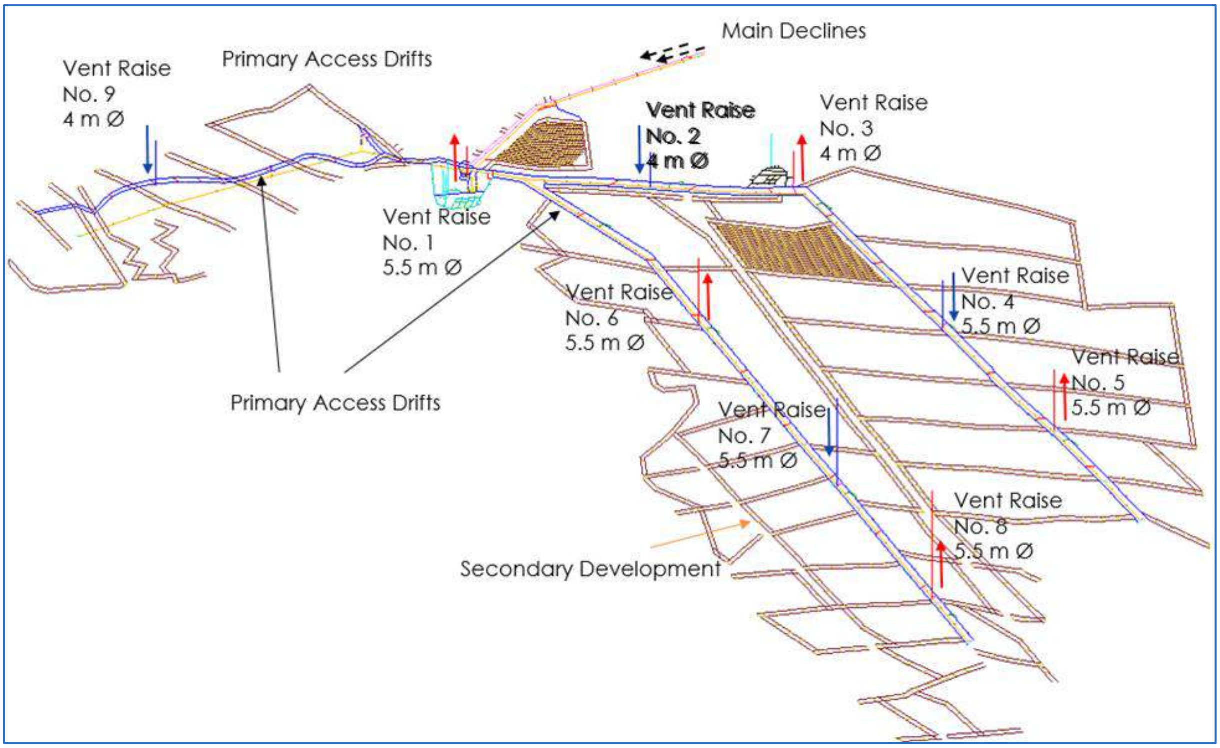


Figure by Stantec, 2017.

The room-and-pillar method will be used in the mineralised zone between 60–150 m, to minimise the risk of surface subsidence. Continuing room-and-pillar mining below 150 m is required in selected areas for production ramp-up. A controlled convergence room-and-pillar test panel will be completed before additional controlled convergence room-and-pillar panels will be approved for mining.

The production development of the room-and-pillar method will be in a grid-like fashion, using 7.0 m wide drifts. Panel sizes were defined using the same criteria as the controlled convergence room-and-pillar method as discussed below. The room development will run parallel to the strike of the panel for dips less than 20°, with belt drives running at an acute angle to the room drifts, to ensure the grade of the production drifts remains at a maximum of 12°. Where the dip is greater than 20°, the rooms will be developed slightly off the strike, to accommodate the acute angle between the room development and the belt drives. Long-term stability is required in the room-and-pillar mining areas to allow access while in production, as the mining front begins at the access and progresses toward the ends of the panel. These room-and-pillar mining areas, designed to prevent subsidence, will remain accessible if maintained and ventilated.

Controlled convergence room-and-pillar mining is based on the strength and strain parameters of the rock that makes up the mining panel supporting pillar or technological pillars and includes the following parameters:

- Ore zone depths below 150 m.
- Strength of the immediate roof (i.e., roof bolting and handling of the rock burst threat).
- Strength and strain parameters of the rocks within the roof of the extraction panel (i.e., the slow bending above the extraction space and in the workings).
- Technological pillars (pillars between rooms) designed to work in the post-destruction strength state to maximise ore extraction.

KGHM CUPRUM Ltd – Research and Development Centre (Cuprum) developed the controlled convergence room-and-pillar methodology (2016, 2017a, 2017b) at its mines in Poland and are the technical contributors to its adaptation for the Project.

Controlled convergence room-and-pillar in the post-destructive state is based on a modified Labasse hypothesis (1949). The pillar height-to-width ratio should be within the range of 0.5–0.8. This ensures the progressive transition of the technological pillars into the post-destructive strength state, enabling a smooth roof-bending strata (destressed and delaminated rock mass) above the workings.

The development schedule focuses on the establishment of necessary mine services and support infrastructure to set up the initial production mining areas and ramp-up to 6 Mtpa ore production and associated development waste. The full production schedule will be based on a 360-day calendar that will be sustained for 17 years with a 26-year LOM.

Mine development is broken down into the following three main phases:

- Phase 1: Development of the Declines to the Main Ore Bins.
- Phase 2: Room-and-Pillar Mining and Controlled Convergence Room-and-Pillar Test Panel.
- Phase 3: Development of Centrale and Sud.

Table 1.15 shows LOM production summary.

Table 1.15 LOM Production Summary

Production by Mining Method	Mined (kt)	Meters (m)	NSR ($/t)	Cu (%)
Ore Development	10,665	114,205	146.20	3.34
Room-and-Pillar	3,397	36,743	243.27	5.29
Controlled Convergence Room-and-Pillar	111,120	813,559	167.86	3.81
Total	**125,182**	**964,507**	**168.06**	**3.81**

The following criteria were applied over the mine life for scheduling purposes.

- Proximity to the Main Accesses and Early Development.
- High-grade and Thickness.
- Ventilation Constraints.
- Mining Direction.
- 300 m Gap Distance between Two Adjacent Panel Fronts.
- Application of a Declining Cut-off Grade.

Using the strategy above, appropriate panels were targeted and scheduled to achieve the highest possible grade profile during ramp-up and full production.

Underground infrastructure involves several components such as ore and waste handling systems, dewatering, maintenance shops, fuelling, ventilation, concrete and shotcrete facilities, refuge stations, etc.

Power will be available from the state-owned utility Société Nationale d'Electricité (SNEL), transmitted at 33 kilovolts (kV) from Kolwezi to the consumer substation located at the mine. Power will be distributed on the mine at 11 kV and 690 volts (V), both on surface and underground. The mine’s maximum demand, including a 20% contingency, is expected to be 38.6 megavolt ampere (MVA) at a power factor of 0.85.

1.16.2 Kamoa 2019 PFS Process

The Kamoa 2019 PFS process plant consists of a 6 Mtpa Run-of-Mine (ROM) concentrator incorporating staged crushing, ball mill grinding and flotation. The output of the process plant is copper concentrate which is sold to external smelters.

Feed will pass through a 300 mm square grizzly underground before being conveyed from the mine to surface stockpiles. An overbelt magnet removes tramp steel from the feed before it is sent to the ROM stockpile.

Four variable-speed apron feeders are available to recover material from the stockpile and feed the crushing plant. ROM is fed onto a 50 mm aperture heavy duty primary screen, from which the oversize is sent to primary crushing and the undersize is sent to secondary crushing.

Primary crusher product joins secondary crusher product and is conveyed to the four sizing screen feed bins. Each bin feeds a sizing screen via a variable speed vibrating feeder. Screen oversize is sent to the three secondary crushing feed bins. Each bin feeds a secondary crusher via a variable speed vibrating feeder.

Sizing screen undersize is sent to the mill feed stockpile. The mill feed stockpile has four vibrating feeders below it that feed onto the two parallel mill feed conveyors.

Milling is conducted in two identical parallel circuits, each consisting of two identical ball mills in series. The primary ball mill grinds to about 150 µm P_{80} and final grinding to 53 µm P_{80} occurs in the secondary mill. The primary ball mill has a ball scats trommel screen and is closed with a cyclone cluster. Cyclone overflow feeds the secondary milling circuit.

The secondary ball mill discharges through a trommel screen to remove ball scats and the trommel undersize gravitates to the mill discharge sump, from where it is pumped to the cyclone cluster. The cyclone overflow feeds the flotation feed conditioning tank while the underflow returns to the secondary ball mill.

Rougher and scavenger flotation takes place in two parallel trains, each of which is a bank of seven cells in series. The first two cells in each train will perform the roughing duty, with the remainder performing scavenger flotation. Rougher concentrate from both trains is pumped direct to a common cleaner and recleaner flotation circuit. Scavenger concentrate is unsuited to direct cleaning and it forms the majority of the regrind mill feed. A common regrind and cleaning circuit treats the scavenger concentrates from both trains. Scavenger tails forms the majority of the final tails stream.

Rougher cleaner concentrate is sent to rougher recleaner flotation and the recleaned rougher concentrate forms the majority of the final copper concentrate. The rougher recleaner concentrate is pumped to the concentrate thickener. All rougher cleaning is open circuit and tails from both the rougher cleaner and rougher recleaner are sent to regrind milling.

The three regrind mill feed streams (scavenger concentrate, rougher cleaner tails and rougher recleaner tails) are pumped to the regrind feed tank. Regrind circuit feed is pumped to the regrind densifying cyclones. Densifying cyclone overflow reports directly to the regrind product tank and cyclone underflow is fed to the regrind mills. Reground material reports to the regrind product tank. The regrind target P_{80} is 10 µm.

Reground material is pumped to the scavenger cleaner flotation conditioning tank. Reagents are added, and the slurry is pumped to the scavenger cleaner flotation bank. Scavenger cleaner concentrate is pumped to scavenger recleaning and scavenger recleaner concentrate is pumped to the concentrate thickener feed tank. Scavenger cleaner and recleaner tails are pumped to the final tailings thickener.

Coarse rougher recleaner concentrate and fine scavenger circuit concentrate combine in the thickener feed tank and feed the thickener by gravity. Thickened concentrate is filtered and then sampled and bagged for transport to customers.

Scavenger tails, scavenger cleaner tails and scavenger recleaner tails combine in the tailings thickener feed tank and flow by gravity to the thickener. All tailings thickener overflow reports to the process water tank. Tailings thickener underflow is pumped to the tailings pumping tank and it is sampled. Multistage slurry pumps deliver the slurry to the tailings storage facility.

1.16.3 Kamoa 2019 PFS Transport

A phased logistics solution is proposed in the Kamoa 2019 PFS. Initially the corridor between southern DRC and Durban in South Africa is viewed as the most attractive and reliable export route. As soon as the railroad between Kolwezi and Dilolo, a town near the DRC-Angolan border, is rehabilitated, production from the Kamoa-Kakula Project is expected to be transported by rail to the port of Lobito in Angola.

1.16.4 Kamoa 2019 PFS Results

The base case described in the Kamoa 2019 PFS is the construction and operation of a stand-alone underground mine, concentrator processing facilities, and associated infrastructure. The base case mining rate and concentrator feed capacity is 6 Mtpa. The PFS is based entirely on the Kamoa 2019 PFS Mineral Reserve.

The Kamoa 2019 PFS maintains the development of the Kamoa Deposit as a stand-alone 6 Mtpa mining and processing complex. The LOM production scenario schedules 125.2 Mt to be mined at an average grade of 3.81% copper, producing 11.4 Mt of high-grade copper concentrate, containing approximately 9.2 billion pounds of copper.

The economic analysis uses a long-term price assumption of US$3.10/lb of copper and returns an after-tax NPV at an 8% discount rate of US$2.3 billion, an increase of 140% compared to the after-tax NPV 8% of US$986 million that was projected in the Kamoa 2017 PFS. The Kamoa 2019 PFS has an after-tax IRR of 26.7% and a payback period of 4.9 years. The LOM average mine site cash cost is US$0.66/lb of copper.

The estimated initial capital cost, including contingency, is US$1.0 billion. The capital expenditure for off-site power, which is included in the initial capital cost, includes a US$64M advance payment to the DRC state-owned electricity company, SNEL, to upgrade two hydropower plants (Koni and Mwadingusha) to provide the Kamoa-Kakula Project with access to clean electricity for its planned operations. Mwadingusha is being upgraded first. The work is being led by Stucky Ltd., of Switzerland; the advance payment will be recovered through a reduction in the power tariff once Kamoa is in operation. The key results of the Kamoa 2019 PFS are summarised in Table 1.16.

Table 1.16 Kamoa 2019 PFS Summary

Item	Unit	Total
Total Processed		
Quantity Milled	kt	125,182
Copper Feed Grade	%	3.81
Total Concentrate Produced		
Copper Concentrate Produced	kt (dry)	11,405
Copper Recovery	%	87.52
Copper Concentrate Grade	%	36.63
Contained Copper in Concentrate	Mlb	9,211
Contained Copper in Concentrate	kt	4,178
Peak Annual Contained Metal in Concentrate	kt	245
10 Year Average		
Copper Concentrate Produced	kt (dry)	487
Contained Copper in Concentrate	kt	178
Mine Site Cash Cost	US$/lb	0.59
Total Cash Cost	US$/lb	1.55
Key Financial Results		
Peak Funding	US$M	1,300
Initial Capital Cost	US$M	994
Expansion Capital Cost	US$M	299
Sustaining Capital Cost	US$M	1,359
LOM Average Mine Site Cash Cost	US$/lb Cu	0.66
LOM Average Total Cash Cost	US$/lb Cu	1.61
Site Operating Cost	US$/t Milled	46.71
After-Tax NPV8%	US$M	2,334
After-Tax IRR	%	26.7
Project Payback Period	Years	4.7
Initial Project Life	Years	26

1.17 Kamoa-Kakula 2019 PEA

The Kamoa-Kakula 2019 PEA analyses a production case with an expansion of the Kakula concentrator processing facilities, and associated infrastructure to 18 Mtpa and includes a smelter and seven separate underground mining operations with associated capital and operating costs. The locations of the seven mines and the boundaries for the PFS and PEA cases are shown in Figure 1.16. The details of the Kamoa-Kakula 2019 PEA are in Section 24). The seven mines ranked by their relative values are:

- Kakula Mine (PFS 6 Mtpa).
- Kansoko Mine (PFS 6 Mtpa).
- Kakula West Mine (PEA 6 Mtpa).
- Kamoa Ouest Mine 1 (PEA 6 Mtpa).
- Kansoko Nord Mine 2 (PEA 6 Mtpa).
- Kamoa Centrale Mine 3 (PEA 6 Mtpa).
- Kamoa Nord Mine 4 (PEA 3 Mtpa).

Figure 1.16 Kamoa-Kakula IDP19 Mining Locations

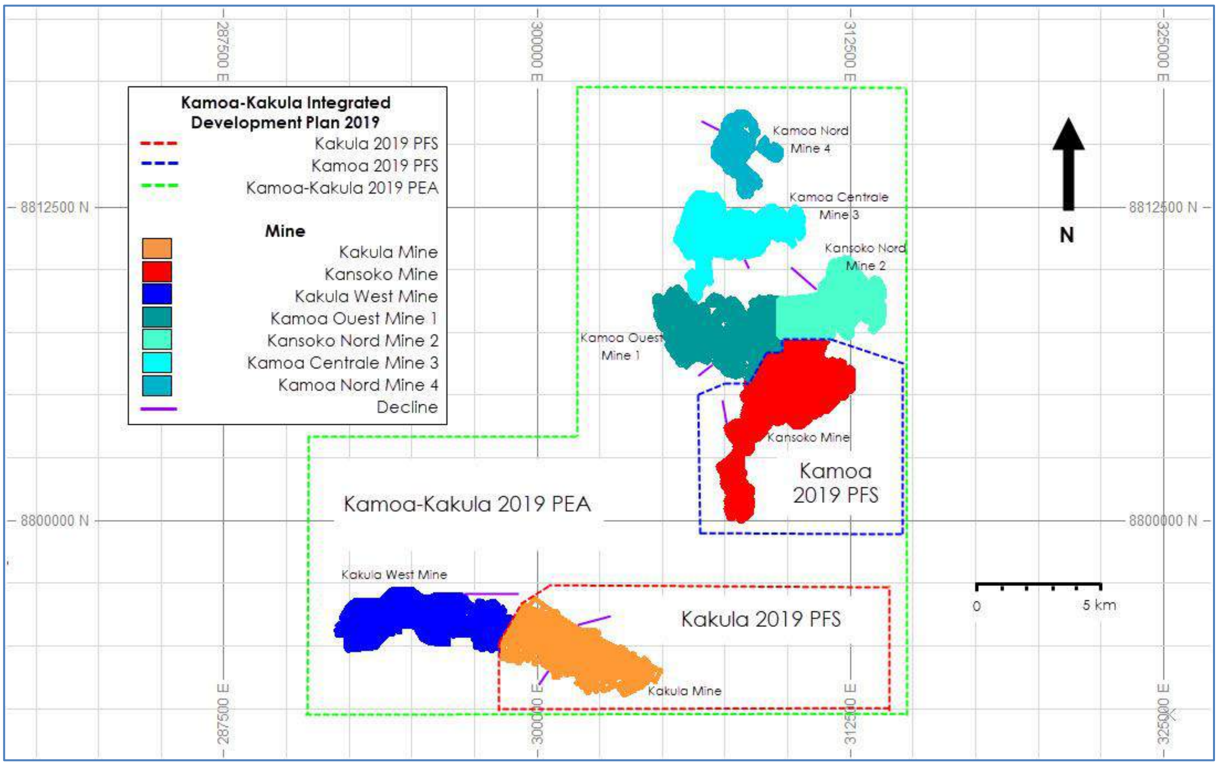


Figure by OreWin, 2019.

The Kamoa-Kakula 2019 PEA is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves – and there is no certainty that the results will be realised. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The Kamoa-Kakula 2019 PEA includes a PEA level study of the Kakula West. Kakula West is separated from Kakula by the West Scarp fault and is planned as an independent mine. The Kakula West, Kamoa Ouest, Kansoko Nord, Kamoa Centrale and Kamoa Nord PEA analyses have been prepared using the Mineral Resources stated in the Kamoa-Kakula 2018 Resource Update. Since that time the Kakula West Mineral Resource has been updated and the updated Mineral Resource has been stated in Section 14 of the Kamoa-Kakula IDP19.

The potential development scenarios at the Kamoa-Kakula Project include the Kamoa-Kakula IDP19 development scenario shown in Figure 1.17. The Kakula decline development is followed by the development of the stoping blocks and construction of the plant. The initial plant capacity of 3 Mtpa is expanded to 6 Mtpa as the Kakula Mine ramps up to full capacity. Following this, the Kansoko Mine is brought into production and the mines continue to ramp up to 12 Mtpa combined by Year 9. The next phase of development described by the Kamoa-Kakula 2019 PEA is from Kakula West to bring total production to 18 Mtpa this is then followed by four additional mines at Kamoa North.

The immediate decisions for Ivanhoe and its partners are to determine the sequence for developing the initial operation.

Figure 1.17 Kamoa-Kakula IDP19 Long-Term Development Scenario

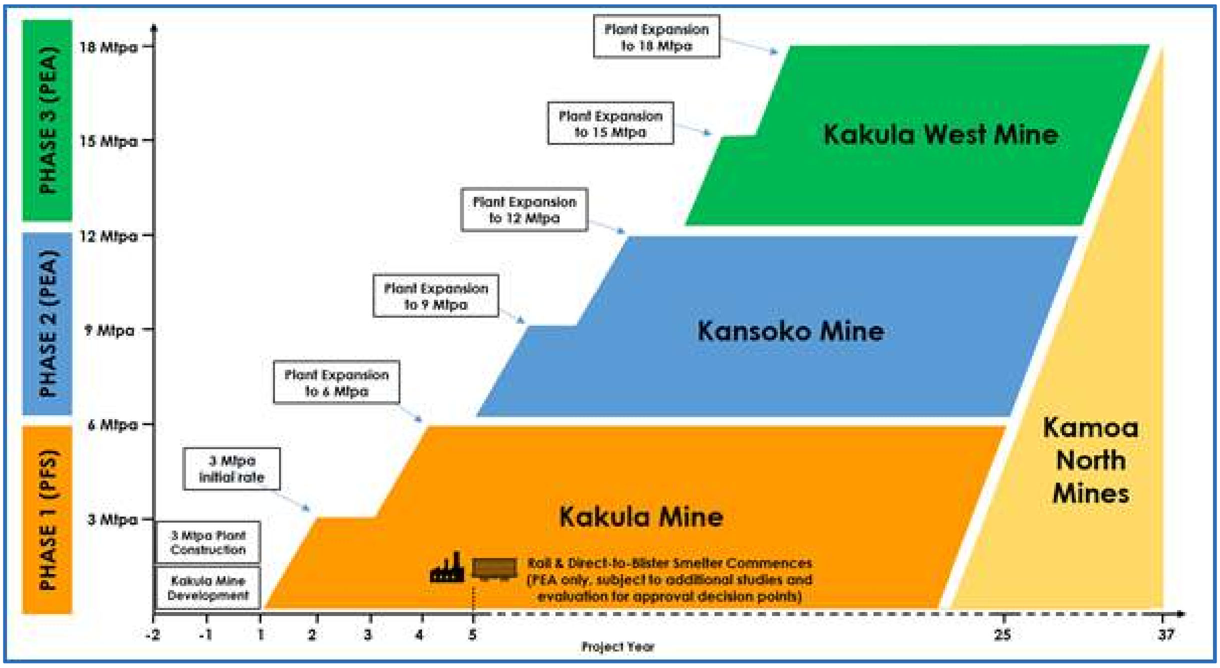


Figure by OreWin, 2019.

A site plan showing the locations of the mines and key infrastructure for Kakula and Kansoko mines is shown in Figure 1.18.

The Kamoa-Kakula 2019 PEA as part of the Kamoa-Kakula IDP19 includes economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability. The results of the Kamoa-Kakula 2019 PEA represent forward-looking information. The forward-looking information includes metal price assumptions, cash flow forecasts, projected capital and operating costs, metal recoveries, mine life and production rates, and other assumptions used in the Kamoa-Kakula 2019 PEA. Readers are cautioned that actual results may vary from those presented. The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially are presented in the body of this report under each relevant section.

Additional studies are required to evaluate feasibility and the timing of increasing plant feed from the Kakula mine, the Kansoko mine and the Kamoa North Mines of the Kamoa Deposit. Also, a sensitivity analysis is required to evaluate feasibility and the timing of an on-site smelter to produce blister copper at the mine site.

Figure 1.18 Kamoa-Kakula Integrated Development Plan 2019 Site Plan

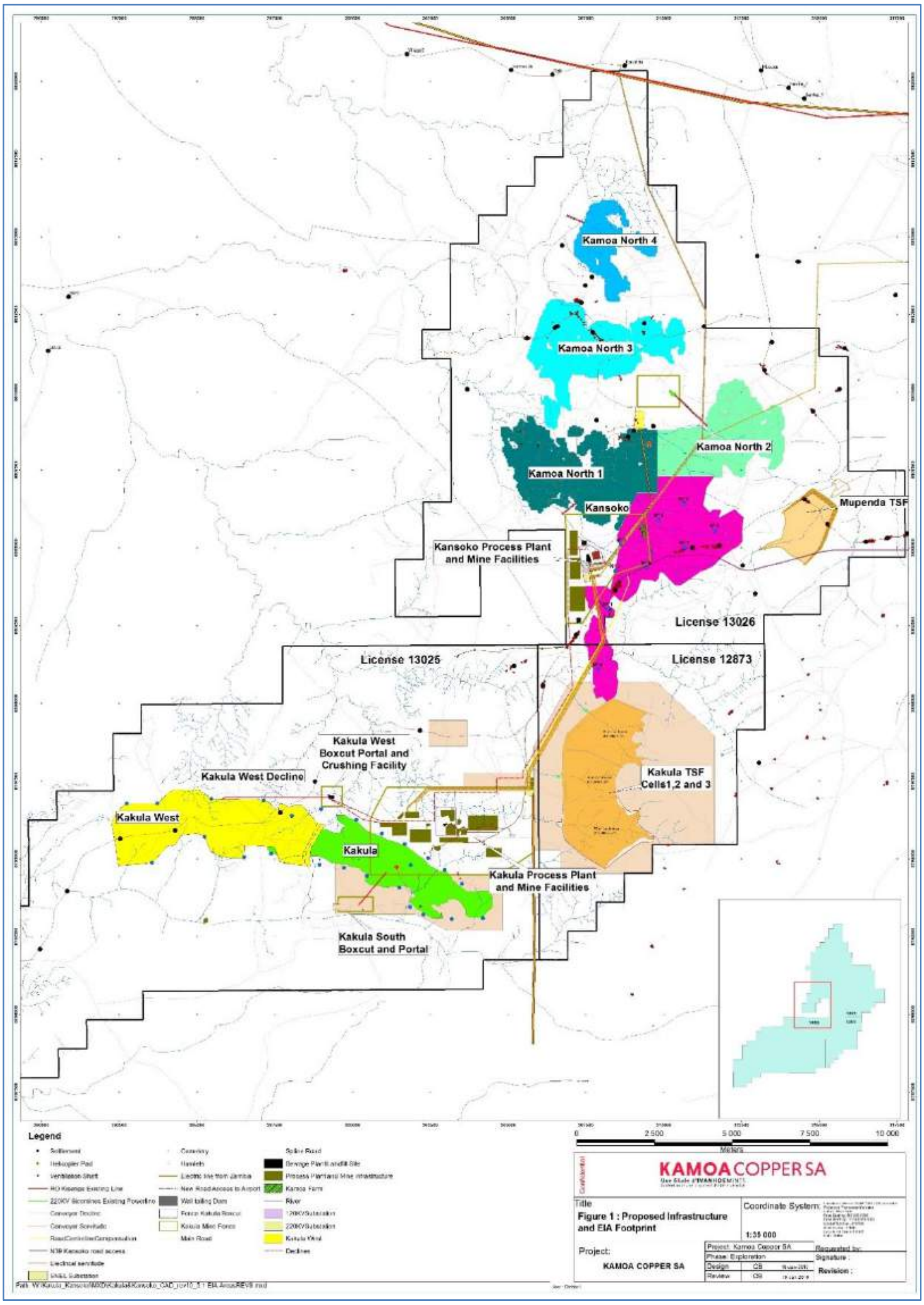


Figure by Kamoa Copper SA, 2019.

1.17.1 Kamoa-Kakula 2019 PEA Results Summary

The Kamoa-Kakula 2019 PEA assesses an alternative development option of mining several deposits on the Kamoa-Kakula Project as an integrated, 18 Mtpa mining, processing and smelting complex, built in three stages. This scenario envisages the construction and operation of three separate mines: first, an initial 6 Mtpa mining operation would be established at the Kakula Mine on the Kakula Deposit; this is followed by a subsequent, separate 6 Mtpa mining operation at the Kansoko Mine using the existing twin declines that were completed in 2017; a third 6 Mtpa mine then will be established at the Kakula West Mine. As the resources at the Kakula, Kansoko and Kakula West mines are mined out, production would begin sequentially at four other mines in the Kamoa North area to maintain throughput of 18 Mtpa to the then existing concentrator and smelter complex.

Each mining operation is expected to be a separate underground mine with a shared processing facility and surface infrastructure located at Kakula. Included in this scenario is the construction of a direct-to-blister flash copper smelter with a capacity of one million tonnes of copper concentrate per annum.

A summary of the key results for the Kamoa-Kakula 2019 PEA scenario are:

- Very-high-grade initial phase of production is projected to have a grade of 7.1% copper in second year of production and an average grade of 5.7% copper over the initial 10 years of operations, resulting in estimated average annual copper production of 386,000 tonnes.
- Annual copper production is estimated at 360,000 tonnes in Year Four.
- Initial capital cost, including contingency, is estimated at US$1.1 billion.
- Average total cash cost of US$0.93/lb of copper during the first 10 years, including sulphuric acid credits.
- After-tax NPV, at an 8% discount rate, of US$10.0 billion.
- After-tax internal rate of return (IRR) of 40.95%, and a payback period of 2.9 years.

The estimated initial capital cost, including contingency, is US$1.1 billion. The capital expenditure for off-site power, which is included in the initial capital cost, includes a US$64M advance payment to the DRC state-owned electricity company, SNEL, to upgrade two hydropower plants (Koni and Mwadingusha) to provide the Kamoa-Kakula Project with access to clean electricity for its planned operations. Mwadingusha is being upgraded first. The work is being led by Stucky Ltd., of Switzerland; the advance payment will be recovered through a reduction in the power tariff.

The Kamoa Kakula 2019 PEA is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves—and there is no certainty that the results will be realised. Mineral Resources do not have demonstrated economic viability and are not Mineral Reserves. Key results of the Kamoa Kakula 2019 PEA are summarised in Table 1.17 and Table 1.18 summarises the overall results. The mining production statistics are shown in Table 1.19. The Kamoa Kakula 2019 PEA 18 Mtpa mill feed and copper grade profile for the LOM are shown in Figure 1.19 and the concentrate and metal production for the LOM are shown in Figure 1.20. Table 1.20 summarises unit operating costs.

Table 1.17 Kamoa-Kakula 2019 PEA Financial Results

Net Present Value (US$M)	Discount Rate	Before Taxation	After Taxation
	Undiscounted	59,520	41,222
	4.0%	28,169	19,240
	6.0%	20,281	13,731
	8.0%	**14,967**	**10,030**
	10.0%	11,280	7,469
	12.0%	8,653	5,651
Internal Rate of Return	–	50.3%	40.9%
Project Payback Period (Years)	–	2.5	2.9

Table 1.18 Kamoa-Kakula 2019 PEA Results Summary for 18 Mtpa Production

Item	Unit	Total
Total Processed		
Quantity Milled	kt	535,217
Copper Feed Grade	%	3.88
Total Concentrate Produced		
Copper Concentrate Produced	kt (dry)	39,039
Copper Recovery	%	85.12
Copper Concentrate Grade	%	45.23
Contained Copper in Concentrate - External Smelter	Mlb	9,930
Contained Copper in Concentrate - External Smelter	kt	4,504
Contained Copper in Blister - Internal Smelter	Mlb	28,559
Contained Copper in Blister - Internal Smelter	kt	12,954
Peak Annual Recovered Copper Production	kt	740
10-Year Average		
Copper Concentrate Produced	kt (dry)	759
Contained Copper in Conc. - External Smelter	kt	121
Contained Copper in Blister - Internal Smelter	kt	261
Mine-Site Cash Cost	US$/lb Cu	0.63
Total Cash Cost	US$/lb Cu	0.93
Key Financial Results		
Peak Funding	US$M	1,099
Initial Capital Cost	US$M	1,078
Expansion Capital Cost	US$M	4,958
Sustaining Capital Cost	US$M	10,811
LOM Average Mine Site Cash Cost	US$/lb Cu	0.86
LOM Average Total Cash Cost	US$/lb Cu	1.10
Site Operating Cost	US$/t Milled	61.47
After-Tax NPV8%	US$M	10,030
After-Tax IRR	%	40.9
Project Payback Period	Years	2.9
Initial Project Life	Years	37

Table 1.19 Kamoa-Kakula 2019 PEA Production and Processing

Item	Unit	Total LOM	Years 1–5	Years 1–10	LOM Average
Total Processed					
Quantity Milled	kt	535,217	4,743	7,915	14,465
Copper Feed Grade	%	3.88	6.68	5.66	3.88
Total Concentrate Produced					
Copper Concentrate Produced	kt (dry)	39,039	478	759	1,055
Copper Concentrate - External Smelter	kt (dry)	8,491	344	211	229
Copper Concentrate - Internal Smelter	kt (dry)	30,549	135	549	826
Copper Recovery	%	85.12	85.62	86.18	85.12
Copper Concentrate Grade	%	45.23	56.76	50.82	45.23
Contained Copper in Concentrate - External Smelter					
Copper	Mlb	9,930	434	266	268
Copper	kt	4,504	197	121	122
Payable Copper in Concentrate - External Smelter					
Copper	Mlb	9,687	424	260	262
Copper	kt	4,394	192	118	119
Contained Copper in Blister - Internal Smelter					
Copper	Mlb	28,559	162	576	772
Copper	kt	12,954	73	261	350
Payable Copper in Blister - Internal Smelter					
Copper	Mlb	28,473	161	574	770
Copper	kt	12,915	73	260	349
Payable Copper					
Copper	Mlb	38,160	586	834	1,031
Copper	kt	17,309	266	378	468

Table 1.20 Kamoa-Kakula 2019 PEA Unit Operating Costs

Item	Payable Copper (US$/lb)		
	Years 1-5	Years 1-10	LOM Average
Mine Site	0.46	0.53	0.75
Smelter	0.04	0.09	0.11
Transport	0.25	0.17	0.15
Treatment and Refining Charges	0.12	0.08	0.08
Royalties and Export Tax	0.18	0.16	0.16
Total Cash Costs	**1.05**	**1.04**	**1.25**
Sulphuric Acid Credits	0.03	0.11	0.15
Total Cash Costs After Credits	**1.02**	**0.93**	**1.10**

Figure 1.19 Kamoa-Kakula 2019 PEA Process Production

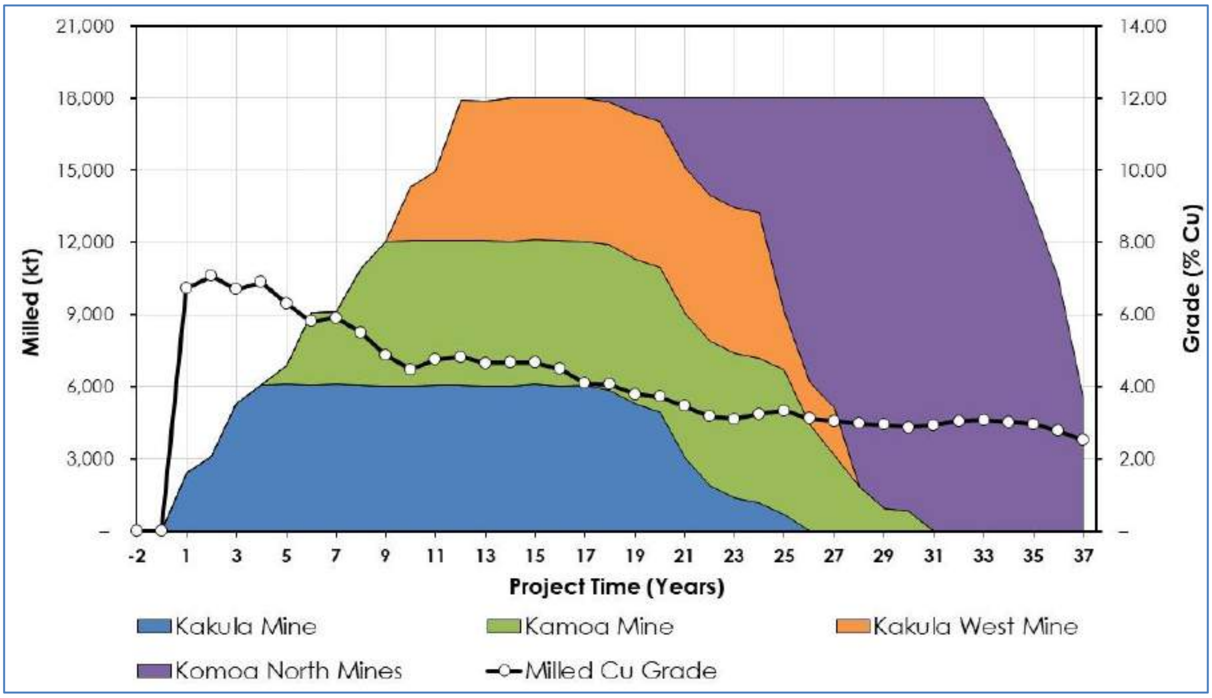


Figure by OreWin, 2019.

Figure 1.20 Kamoa-Kakula 2019 PEA Concentrate and Metal Production

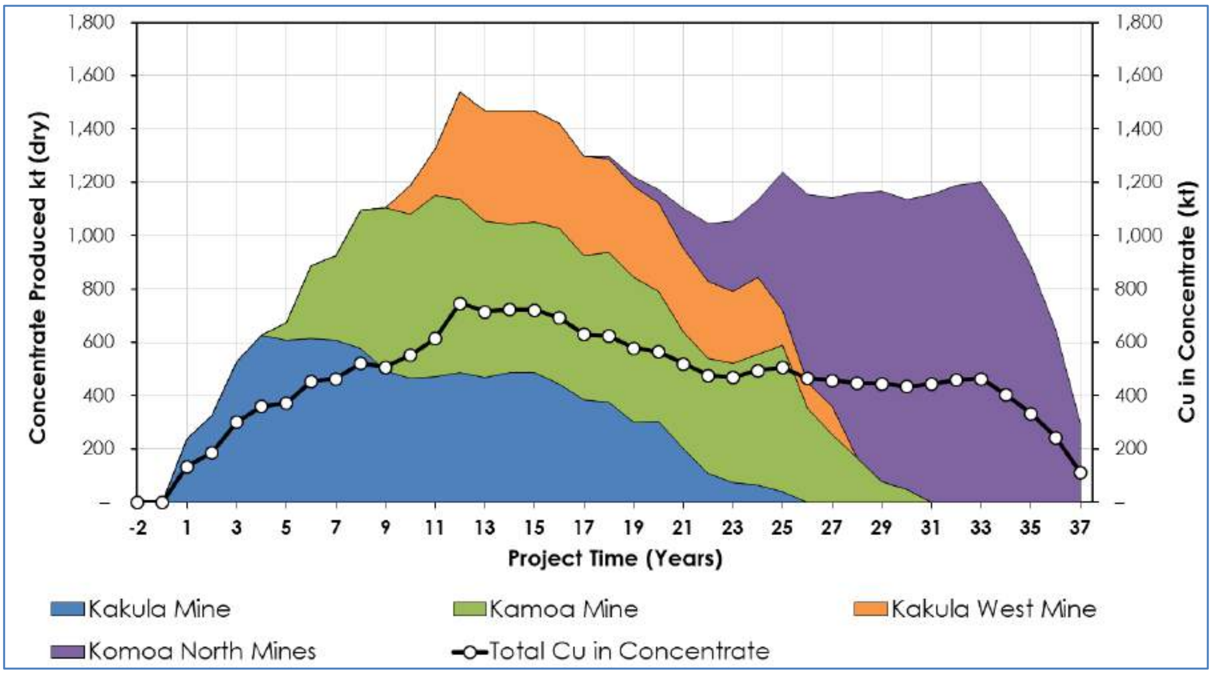


Figure by OreWin, 2019.

Table 1.21 provides a breakdown of revenue and operating costs. Capital costs for the project are detailed in Table 1.22.

Table 1.21 Kamoa-Kakula 2019 PEA Revenue and Operating Costs

	Total LOM US$M	Years 1–5	Years 1–10	LOM Average
		US$/t Milled		
Revenue				
Copper in Blister	88,266	105.43	224.77	164.92
Copper in Concentrate	30,029	277.36	101.89	56.11
Acid Production	5,786	3.46	11.25	10.81
Gross Sales Revenue	**124,081**	**386.25**	**337.90**	**231.83**
Less: Realisation Costs				
Transport	5,847	31.40	17.96	10.92
Treatment and Refining	2,899	14.82	8.45	5.42
Royalties and Export Tax	6,103	22.69	16.96	11.40
Total Realisation Costs	**14,848**	**68.90**	**43.36**	**27.74**
Net Sales Revenue	**109,233**	**317.34**	**294.54**	**204.09**
Site Operating Costs				
Underground Mining	18,513	35.12	35.29	34.59
Processing	7,350	15.27	14.97	13.73
Tailings	107	0.26	0.23	0.20
Smelter	4,117	4.52	9.99	7.69
General and Administration	1,750	6.25	5.32	3.27
SNEL Discount	-194	-3.11	-2.38	-0.35
Customs Duties	1,254	2.79	2.54	2.34
Total	**32,897**	**61.09**	**65.96**	**61.47**
Net Operating Margin	**76,337**	**256.25**	**228.58**	**142.62**
Net Operating Margin	**69.88%**	**80.75%**	**77.60%**	**69.88%**

Totals have been rounded.

Table 1.22 Kamoa-Kakula 2019 PEA Capital Costs

Description	Initial Capital US$M	Expansion Capital US$M	Sustaining Capital US$M	Total US$M
Mining				
Underground Mining	287	753	6,216	7,256
Capitalised Preproduction	107	–	–	107
Subtotal	**394**	**753**	**6,216**	**7,364**
Power and Smelter				
Smelter Total	–	770	716	1,487
Power Supply Off Site	64	–	–	64
Subtotal	**64**	**770**	**716**	**1,551**
Concentrate and Tailings				
Process Plant	190	863	950	2,003
Tailings	24	53	167	244
Subtotal	**214**	**916**	**1,116**	**2,246**
Infrastructure				
Plant Infrastructure	109	756	707	1,573
General Infrastructure	–	353	204	557
Contractor's and Owner's Camps	–	–	73	73
Rail Link	–	72	–	72
Subtotal	**109**	**1,181**	**985**	**2,275**
Indirects				
EPCM	56	147	93	297
Owners Cost	103	323	–	426
Customs Duties	29	141	326	496
Closure	–	–	362	362
Subtotal	**188**	**611**	**782**	**1,581**
Capital Expenditure Before Contingency	**968**	**4,233**	**9,816**	**15,017**
Contingency	110	725	995	1,831
Capital Expenditure After Contingency	**1,078**	**4,958**	**10,811**	**16,847**

Totals have been rounded.

Figure 1.21 compares the reported copper production in 2025 for the 20 highets producers by paid copper production. The Kamoa-Kakula 2019 PEA production is from the projected peak copper production which occurs in Year 12. Figure 1.22 shows the nominal paid copper production and head grade of the world 10 largest new greenfield projects. The estimates are based on public disclosure and information gathered by Wood Mackenzie. The Kamoa-Kakula IDP19 was not reviewed by Wood Mackenzie prior to filing.

Figure 1.21 2025 Predicted World Copper Producer Production

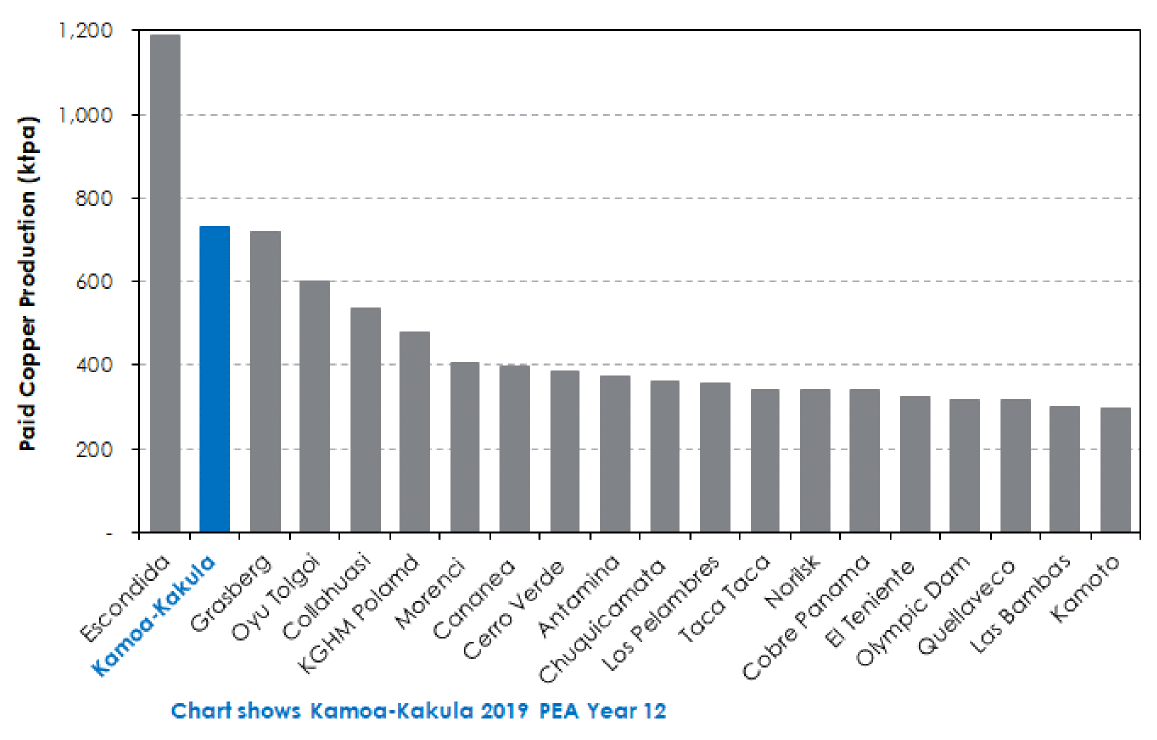


Figure by Ivanhoe, 2019. Source: Wood Mackenzie.

Figure 1.22 World Copper Producer Production and Head Grade

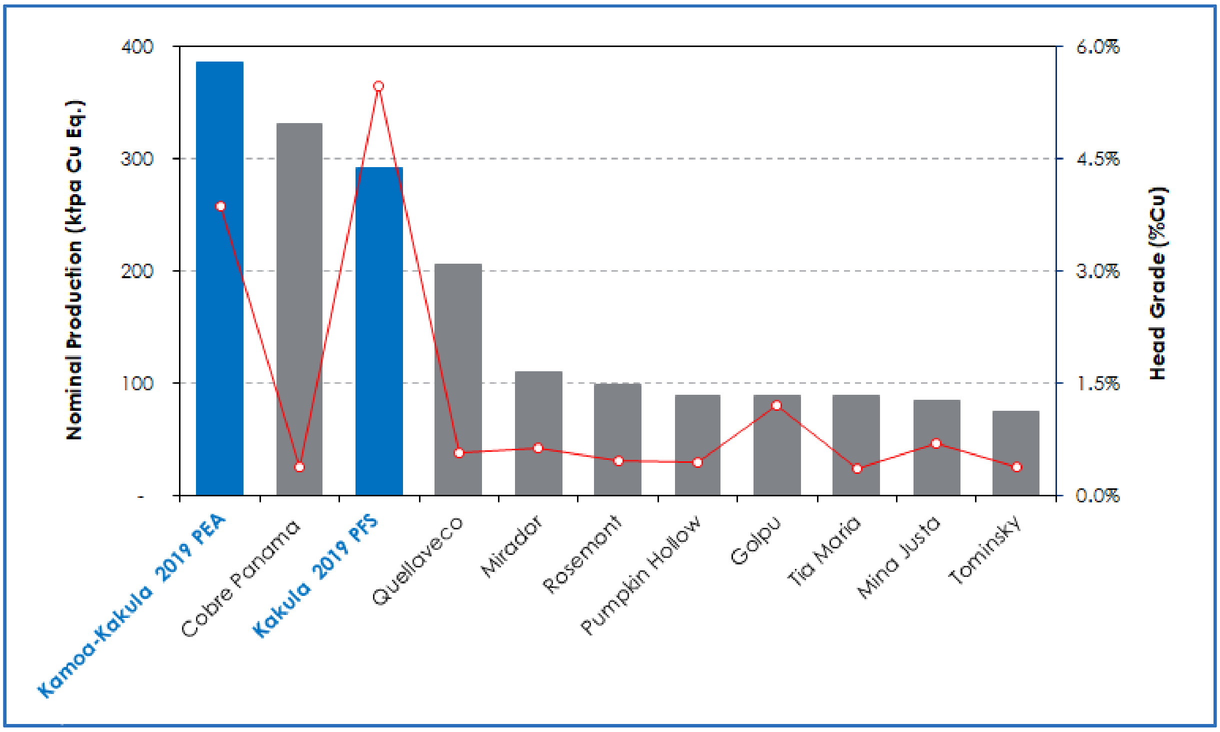


Figure by Ivanhoe, 2019. Source: Wood Mackenzie.

The after-tax NPV sensitivity to metal price variation is shown in Table 1.23 for copper prices from US$2.00/lb to US$4.00/lb.

The annual and cumulative cash flows are shown in Figure 1.23 (annual cash flow is shown on the left vertical axis and cumulative cash flow on the right axis).

Table 1.23 Kamoa-Kakula 2019 PEA Copper Price Sensitivity

After-Tax NPV (US$M)	**Copper Price - US$/lb**					
Discount Rate	**2.00**	**2.50**	**3.00**	**3.10**	**3.50**	**4.00**
Undiscounted	13,117	25,902	38,668	41,222	51,435	64,154
4.0%	5,684	11,850	18,008	19,240	24,165	30,307
6.0%	3,788	8,311	12,828	13,731	17,341	21,845
8.0%	2,510	5,931	9,347	**10,030**	12,758	16,164
10.0%	1,627	4,286	6,939	7,469	9,587	12,231
12.0%	1,005	3,120	5,229	5,651	7,332	9,433
15.0%	385	1,942	3,493	3,803	5,037	6,579
IRR	18.0%	28.9%	39.0%	**40.9%**	48.3%	57.0%

Figure 1.23 Kamoa-Kakula 2019 PEA Projected Cumulative Cash Flow

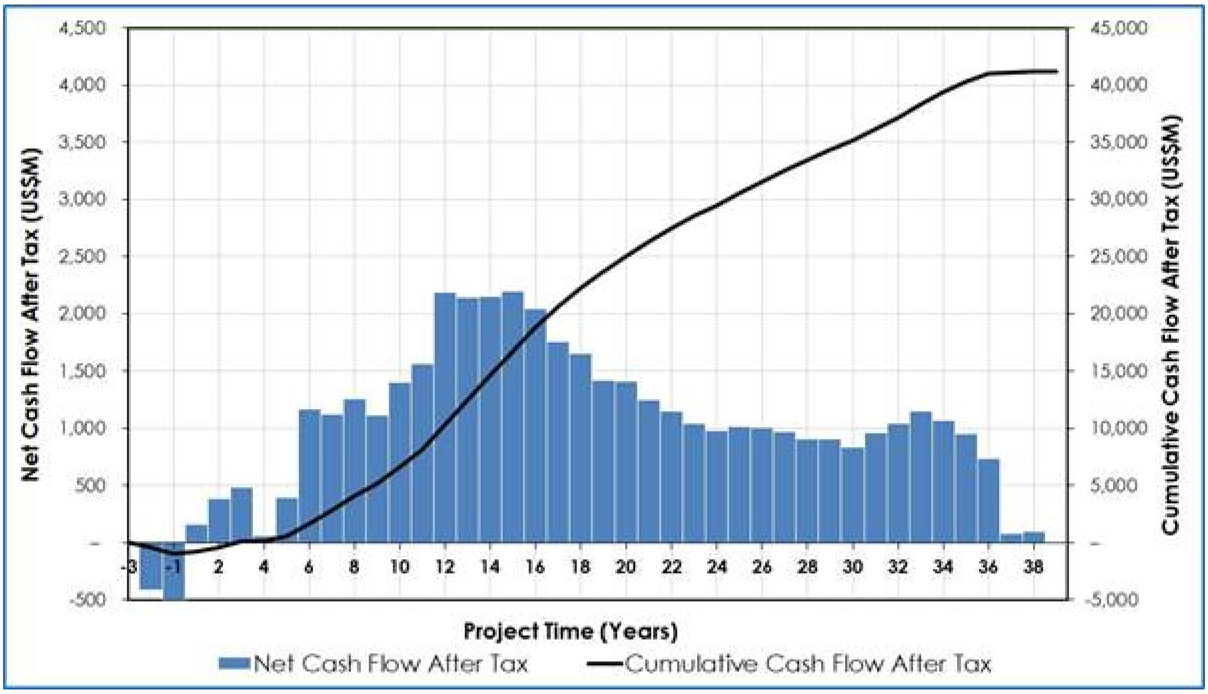


Figure by OreWin, 2019.

1.17.2 Kamoa-Kakula 2019 PEA Mining

Mining methods in the Kamoa-Kakula 2019 PEA are assumed to be a combination of the controlled convergence room-and-pillar mining method, drift-and-fill with paste fill mining method, and room-and-pillar mining method. At Kakula Mine the mining method is drift-and-fill as described in the Kakula 2019 PFS, at the Kansoko Mine the mining method is controlled convergence room-and-pillar method. At Kakula West there is a combination of drift-and-fill and controlled convergence room-and-pillar. Selection of the mining method was dictated by mining height and dip. The controlled convergence room-and-pillar method was selected for heights greater than 3 m and less than 6 m, and dip less than 25°. The drift-and-fill with paste fill was selected for heights greater than 6 m. The drift-and-fill with paste fill method was also selected for heights greater than 3 m and less than 6 m, and dip greater than 25°. At the Kamoa North Mines the mining method selected is controlled convergence room-and-pillar. The Kamoa-Kakula 2019 PEA is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves—and there is no certainty that the results will be realised. Mineral Resources do not have demonstrated economic viability and are not Mineral Reserves.

1.17.3 Kamoa-Kakula 2019 PEA Processing and Infrastructure

The Kamoa-Kakula 2019 PEA scenario assumes that the project proceeds with first completing the Kakula 6 Mtpa process plant, built in two stages of 2 x 3 Mtpa, followed by an expansion to include the 6 Mtpa Kamoa concentrator plant to take production to 12 Mtpa. A final expansion of a third 6 Mtpa concentrator stream is included to take the central complex's processing capacity to 18 Mtpa. The Kamoa-Kakula 2019 PEA has processing and infrastructure facilities that include:

- A 18 Mtpa central processing facility complete with surface crushing and screening, milling, and flotation, consisting of three 6 Mtpa concentrator streams, a smelter, and associated infrastructure located at the Kakula Mine area.
- The Kakula Mine and dedicated surface infrastructure on the Kakula Deposit.
- The Kansoko Mine on the Kansoko Sud and Kansoko Centrale areas of the Kamoa Deposit; including associated overland conveying systems.
- Dedicated surface infrastructure including associated overland conveying systems at Kakula West Mine and Kamoa North Mines 1 to 4.

The Kakula process plant will be the first of three 6 Mtpa circuits to be located at the central processing complex. The Kakula concentrator (Central Complex Concentrator 1) includes a 15,000 t ROM stockpile to feed a 6 Mtpa Run-of-Mine (ROM) concentrator based on staged crushing and screening, followed by two stage series, ball milling. The ball milling product is upgraded in the flotation circuit which is designed to produce two different concentrate product, i.e. a high grade and a medium grade product. These two concentrate products are combined to form the final concentrate.

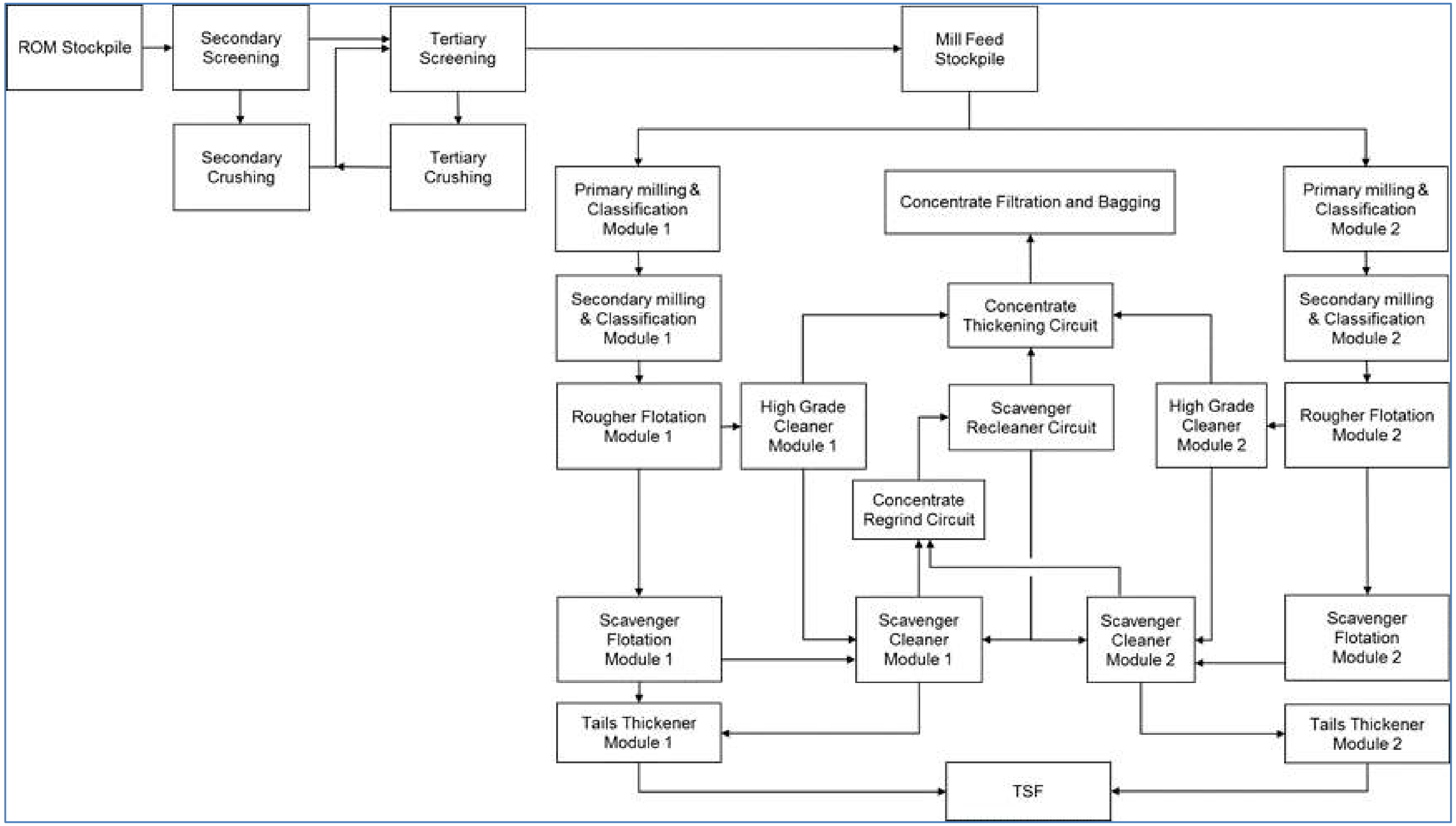
The Kakula design allows the Central Complex Concentrator 1 to be built into two phases in order to be aligned with the mine production schedule. Phase 1 will treat 3 Mtpa in line with the mine ramp up and the throughput will be doubled during Phase 2 to 6 Mtpa. Refer to Section 17 for more detail on the Kakula design. A high-level block flow diagram of the Central Complex Concentrator 1 is presented in Figure 1.24.

Following the ramp-up of Central Complex Concentrator 1 to 6 Mtpa, the complex will be expanded by the addition of the Central Complex Concentrator 2 at the Kakula Mine Area. Central Complex Concentrator 2 will be based on the Kansoko circuit design. The expansion from 6 Mtpa to 12 Mtpa will also be completed in a two phased approach, as dictated by the mining plan.

Concentrate will be conveyed from the adjacent concentrator complex into a concentrate shed located in the smelter complex. The smelting process utilizes the direct-to-blister smelting technology (DBF), which is proven for treating high copper, low sulphur copper concentrates similar to those envisaged for the Kamoa-Kakula project. Copper concentrate is first dried in a steam dryer before being oxized with oxygen enriced air in the reaction shaft of the DBF to produce blister copper and SO_2 offgas in a single stage flash smelting process. The SO_2 laden offgases are dedusted and sent to a double-contact-double adsorption sulphuric acid plant for production of high strength acid, which is sold to the local market. Copper is recovered from the DBF slag in a downstream electric slag cleaning via reduction with metallurgical coke to produced blister copper. Electric furnace slag still containing up to 4% copper, is slowed cooled, crushed and sent to a slag flotation plant to recovery the residual copper, in the form of concentrate, which is then back to the concentrate storage shed and mixed with fresh concentrate. The final tailings from the slag plant containing 0.8% copper will be pumped to the central complex concentrator tailings facility.

The smelter is designed to process 1,000 ktpa concentrate, producing up to 467 ktpa blister copper and an average 710 ktpa of high strength sulphuric acid.

Figure 1.24 Central Complex Concentrator 1 Block Flow Diagram



1.18 Interpretation and Conclusions

1.18.1 Kamoa-Kakula Integrated Development Plan 2019

The Kakula 2019 PFS has identified a Mineral Reserve and development path that has confirmed significant value in the Kakula Deposit. Kakula and Kansoko studies are at the same PFS level, and the studies indicate that Kakula will have a greater value than Kansoko.

The Kamoa-Kakula 2019 PEA indicates that there is potential value in a central processing facility, on-site smelting and expansions in production. In order to identify this potential, further study will be needed using a whole of project approach into the long-term options to maximise the efficient extraction of the Kamoa-Kakula Mineral Resources.

1.18.2 Mineral Resource Estimate

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). Amec Foster Wheeler has checked the data used to construct the resource models, the methodology used to construct it (Datamine macros), and has validated the resource models. Amec Foster Wheeler finds the Kamoa and Kakula resource models to be suitable to support prefeasibility level mine planning.

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

- Drill spacing.
    - The drill spacing at the Kamoa and Kakula deposits is insufficient to determine the effects of local faulting on lithology and grade continuity assumptions. Local faulting could disrupt the productivity of a highly-mechanised operation. In addition, the amount of contact dilution related to local undulations in the SMZ has yet to be determined for both deposits. Ivanhoe plans to study these risks with the declines currently in progress at Kamoa and Kakula.
    - Delineation drill programs at the Kamoa deposit will have to use a tight (approximately 50 m) spacing to define the boundaries of mosaic pieces (areas of similar stratigraphic position of SMZs) in order that mine planning can identify and deal with these discontinuities. At the Kakula deposit, the mineralisation appears more continuous compared to Kamoa.
- Assumptions used to generate the data for consideration of reasonable prospects of eventual economic extraction for the Kamoa deposit.
    - Mining recovery could be lower and dilution increased where the dip locally increases on the flanks of the domes. The exploration decline should provide an appropriate trial of the conceptual room-and-pillar mining method on the Kamoa deposit in terms of costs, dilution, and mining recovery. The decline will also provide access to data and metallurgical samples at a bulk scale that cannot be collected at the scale of a drill sample.
- Assumptions used to generate the data for consideration of reasonable prospects of eventual economic extraction for the Kakula deposit.
    - A controlled convergence room-and-pillar technique is being studied which provides the opportunity for reduced costs.

- Exploitation of the Kamoa-Kakula Project requires building a greenfields project with attendant 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 Kamoa and Kakula Mineral Resource estimates.
- Commodity prices and exchange rates.
- Cut-off grades.

1.18.3 Mineral Reserve Estimation

Mineral Reserves for the Kamoa 2019 PFS and Kakula 2019 PFS conform to the requirements of CIM Definition Standards (2014).

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

- The testing of the controlled convergence room-and-pillar mining method to the Kamoa deposit.
- The security and timeliness of binder supply to the paste plant at Kakula.
- Any changes to the resource model as a result of further definition drilling at the site.
- The availability of reliable power to the site.
- Commodity prices and exchange rates.

1.19 Recommendations

1.19.1 Further Assessment

Ivanhoe now has three areas within the Kamoa-Kakula Project (Kamoa, Kakula and Kakula West) that warrant further assessment and are at different stages of study and development. Kakula is a very high-grade Mineral Resource that is separate to Kamoa and could be developed as a separate mine and processing facility, and given this, further study should be undertaken. The Kamoa-Kakula 2019 PEA has identified potential development scenarios for Kamoa and Kakula deposits that suggest expansion of the initial project. The next phase of detailed study should be to prepare a feasibility study on Kakula. A whole of project approach should be undertaken to optimise the project and to take the project through the study phases to production. The key areas for further studies are:

- The Kakula 2019 PFS has identified a Mineral Reserve and development path for the Kakula Deposit. It is recommended that Kakula be progressed to a feasibility study.
- The Kamoa-Kakula 2019 PEA indicates that there is potential value in a central processing facility, on-site smelting and expansions in production. In order to identify this potential, further study will be needed. It is recommended that these studies are undertaken using a whole of project approach into the long-term options to maximise the efficient extraction of the Kamoa-Kakula Mineral Resources.
- Rail and power options for the Project remain important considerations and studies to increase the confidence in the assumptions should continue.
- Continue to monitor the regulatory provisions to be adopted, ensuring as far as possible, continued adequate adherance to the relevant legislative requirements.
- Revisions and updates of the long-term whole of project planning as the Mineral Resources are further defined. Including expanding and optimising the project production rate by considering concentrator and smelter capacities that are matched to the power supply availability, mine production and transport options.
- Other mining areas and additional mines from the Kamoa deposit.
- Rail transport to Lobito or potential new road alternatives.
- Continue infill drilling programme to upgrade resource categorisation, enhance geotechnical database and its application to mine design and ground support, and better understand the continuity of the deposit and impacts on productivities and dilution.
- Consider an underground exploration programme at Kakula to attain first-hand information on actual mining conditions and to validate design assumptions.
- Complete hydrological studies and data evaluation to better determine impacts on underground mining conditions and productivities.

1.19.2 Drill Programme

Extensive drilling has been completed at Kakula and Kakula West, with definition of an Indicated Mineral Resource of sufficient size to support stand-alone mining operations at Kakula and Kakula West. Whilst additional drilling is required to define the edge of the high-grade mineralisation in some areas, and limited ongoing infill drilling occurs at Kakula West, the aims of the drill programme have been achieved, and drilling is not planned to continue at similar rates in 2019 compared to the previous three years. Additional exploration drilling planned to test targets elsewhere within the Project is planned, as well as ongoing drilling at Kamoa North, where high-grade corridors are still being defined. The drill plan is expected to adjust as ongoing results become available. Amec Foster Wheeler has recommended a total 2019 programme of 55,000 m at a cost of $9.0M.

1.19.3 Processing Plant

Relative proportions of the major copper minerals, which are chalcopyrite, bornite and chalcocite, are variable throughout each of the Kamoa project deposits. However, chalcocite dominance is a unique characteristic of Kakula that reduces its mineralogical variability compared to other mineralised zones in the Kamoa project. Another factor reducing the relative mineralogical variability at Kakula is a lack of surface-oxidation related supergene.

The supergene mineralisation in the Kamoa and Kansoko deposits, especially the supergene associated with surface weathering, is characterised by variable proportions of non-floating copper minerals such as malachite, cuprite and native copper. In some supergene intersections, the acid soluble copper (ASCu) proportion can be as high as 90% of total copper. Other supergene areas have ASCu proportions as low as 10%, a level in the same range as the fresh hypogene mineralisation. The value of using %ASCu as a proxy for the degree of supergene alteration has not yet been quantified. However, when the ASCu proportion is as high as 90% it can be assumed that recovery potential by sulphide flotation will be minimal.

Assuming a consistent level of silica contamination can be maintained by the flotation cleaning steps, copper concentrate grade is determined by the copper sulphide mineral mix. Therefore, prediction of copper concentrate grade requires knowledge of the relative copper sulphide mineral proportions in the feed, as all these minerals are recovered at levels greater than 90% to concentrate. If a concentrate is prepared for sale to a third-party smelter, then control of concentrate grade (and its future prediction) is important but not critical. In the case where a smelter is constructed on site at Kamoa, the grade of concentrate, and its future prediction, become critical because the smelter feed quality must be controlled within strict parameters. Any future studies considering smelting on site must incorporate an allowance for testwork to determine which grade control measurements are required to provide a high level of concentrate grade predictability.

Copper recovery from hypogene mineralisation is determined by the amount of copper locked in the non sulphide gangue (NSG, such as quartz). Test work to date has shown this to be consistently in the order of 10% to14% of total copper regardless of sulphide mineralogy. This results in ultimate copper recoveries in the range 86% to 90% for Hypogene samples. Consequently, there is little difficulty in making recovery predictions for hypogene samples together with samples from the deeper supergene zones where almost all copper mineralisation is sulphide and recoverable. The majority of the ores at Kamoa and Kansoko and all of the ore at Kakula are hypogene or in supergene-categories where recovery can be predicted.

Copper recovery from supergene mineralisation associated with surface oxidation is much more difficult to predict. Making such predictions requires an estimate of the proportion of non-floating copper minerals like malachite, cuprite and native copper. Generally, an ASCu assay will provide a reasonable estimate of non-floating copper proportion, provided the ASCu is greater than about 15% of total copper. At less than 15% ASCu a large proportion of the dissolving copper is likely to be floatable, because the hypogene ASCu grades average about 12% of total copper while all copper in hypogene can be considered flotable. It is recommended that where the ASCu is >15% of total copper in Kansoko ore, then a reduced recovery prediction method be applied, specific to surface-oxidised supergene mineralisation. Currently, the recovery prediction for this specific mineralisation class is useful, but requires more verification and refinement before it can be relied upon in a production situation.

It is the opinion of the Process QP that the dominance of the hypogene and deep supergene mineralised types in the project mean that the problems with predicting supergene recoveries are not material to any of the PEA or PFS scenarios. A lack of accurate prediction of copper concentrate quality from feed mineralogy could have material production effects in the scenario where a smelter is constructed as part of the project. However, sufficient time exists after commencement of the project to implement a predictive method ahead of the currently envisaged smelter implementation. Lack of an accurate grade and quality prediction is not a material issue for concentrate sales scenarios, provided the customer's copper grade specification windows are reasonable.

2 INTRODUCTION

2.1 Ivanhoe Mines Ltd

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 (the 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, including a land package of ~9,000 km^2 in the Central African Copperbelt with drill-ready grass-roots prospects.

The original Kamoa copper deposit discovery was made by Ivanplats Limited, which subsequently changed its name to Ivanhoe Mines Ltd. in 2013. For the purposes of this Report, the name "Ivanhoe" refers interchangeably to Ivanhoe's predecessor companies, Ivanplats Limited, Ivanhoe Nickel and Platinum Ltd., and the current subsidiary companies. Advancing the Kamoa-Kakula and Platreef Projects from discovery to production is a key near-term objective.

Ivanhoe owns a 49.5% share interest in Kamoa Holding Limited (Kamoa Holding), an Ivanhoe-Zijin subsidiary that presently owns 80% of the Kamoa-Kakula Project. Zijin owns a 49.5% share interest in Kamoa Holding. The remaining 1% interest in Kamoa Holding is held by privately-owned Crystal River Global Limited.

A 5%, non-dilutable interest in Kamoa Copper was transferred to the DRC government on 11 September 2012, for no consideration, pursuant to the DRC Mining Code. On 11 November 2016, the DRC Minister of Mines and Minister of Portfolio, Ivanhoe, and Zijin Mining Group Co., Ltd., signed an agreement that transfers an additional 15% interest in the Kamoa-Kakula Project to the DRC government, increasing its total stake in the Project to 20%. As a result of the transaction, Ivanhoe and Zijin each hold an indirect 39.6% interest in the Kamoa-Kakula Project, while Crystal River Global Limited holds an indirect 0.8% interest, and the DRC Government holds a direct 20% interest in the Kamoa-Kakula Project.

2.2 Terms of Reference

The Kamoa-Kakula IDP19 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 for the Project located in the DRC.

The Project is situated in the Kolwezi District of Lualaba Province, DRC. The Project is located within the Central African Copperbelt, approximately 25 km west of the provincial capital of Kolwezi and about 270 km west of the regional centre of Lubumbashi. The Project includes the Kamoa and Kakula stratiform copper deposits. The declines providing the initial mine access into these deposits are approximately 11 km apart.

The following companies have undertaken work in preparation of IDP19:

- OreWin: Overall Report preparation, Kakula 2019 PEA analyses, Kakula and Kamoa North underground mining, Kamoa-Kakula combined production schedules, and financial models.
- Amec Foster Wheeler: Geology, drillhole data validation, and Mineral Resource estimation for Kamoa and Kakula.
- SRK Consulting Inc.: PFS Mine geotechnical recommendations.
- Stantec and KGHM Cuprum: Kamoa 2019 PFS and Kakula 2019 PFS underground mine planning and design.
- Golder Associates: Paste backfill, hydrology, hydrogeology, and geochemistry.
- DRA Global: Process and infrastructure.
- Epoch Resources (Pty) Ltd: Tailings Storage Facility (TSF).
- Kamoa Copper SA: Property description and location, ownership, mineral tenure, environmental studies, permitting and social and community and marketing.

This Report uses metric measurements. The currency used is U.S. 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 to 1.5, 1.15, 1.15.3, 1.15.4, 1.16, 1.16.3, 1.16.4, 1.17, 1.17.2, 1.18, 1.18.1, 1.19, 1.19.1; Section 2; Section 3; Section 4; Section 5; Section 19; Section 20; Sections 21.1, 21.1.3, 21.2, 21.2.1; 21.2.8, 21.3; Section 22; Section 23; Sections 24.1, 24.2, 24.3, 24.4, 24.4.2, 24.5, 24.7; Sections 25.2; Section 26.1; Section 27.
- Dr. Harry Parker, SME Registered Member (2460450), Consulting Mining Geologist and Geostatistician, Amec Foster Wheeler E&C Services Inc. (Amec Foster Wheeler), a Wood plc company (Wood) was responsible for: Sections 1.6 to 1.10, 1.12, 1.13, 1.18.2, 1.19.2; Section 2; Section 3; Section 6; Section 7; Section 8; Section 9; Sections 10.1 to 10.6, 10.9, 10.10; Sections 11.1 to 11.3, 11.5 to 11.12; Section 12; Section 14; Section 25.1; Section 26.2; Section 27.
- Gordon Seibel, SME Registered Member (2894840), Principal Geologist, Amec Foster Wheeler, a Wood company, was responsible for: Sections 1.6 to 1.10, 1.12, 1.13, 1.18.2, 1.19.2; Section 2; Section 3; Section 6; Section 7; Section 8; Section 9; Sections 10.1 to 10.6, 10.9, 10.10; Sections 11.1 to 11.3, 11.5 to 11.12; Section 12; Section 14; Section 25.1; Section 26.2; Section 27.
- William Joughin, FSAIMM (55634), employed by SRK Consulting (South Africa) (Pty) Ltd as Principal Consultant, was responsible for: Section 2; Section 10.7; Section 11.3; Section 16.1; Section 24.4.1
- Jon Treen P. Eng (Mining), PEO (90402637), employed by Stantec Consulting International LLC as Mining Business Line Leader, was responsible for: Sections 1.14, 1.15.1, 1.16.1, 1.18.3; Section 2; Section 3; Section 15; Section 16.2; Section 21.1.1, 21.2.2, 21.3; Section 25.3; Section 26.3; Section 27.
- Marius Phillips, MAusIMM (CP 227570), Vice President Process, DRA Global, was responsible for: Sections 1.11, 1.15.2, 1.16.2, 1.17.3, 1.19.3; Section 2; Section 10.8; Section 11.4; Section 13; Section 17; Sections 21.2.3, 21.3; Sections 24.6; Section 25.4 ; Section 26.4; Section 27.
- Gregory Ruiter Pr. Eng, ECSA (980275), Vice President Studies, DRA Global. was responsible for: Section 2; Section 3; Section 18; Sections 21.1.3, 21.2.4 to 21.2.7, 21.3; Section 24.6; Section 25.5 ; Section 26.5; Section 27.

2.4 Site Visits and Scope of Personal Inspection

Site visits were performed as follows:

Mr. Bernard Peters visited the site from 15 February 2010 to 17 February 2010, from 27 April 2010 to 30 April 2010, on 15 November 2012, from 12 September 2015 to 14 September 2015, from 24 October 2016 to 25 October 2016, on 28 June 2017 to 29 June 2017 and from 6 August 2018 to 8 August 2018. The site visits included briefings from Ivanhoe Mine Ltd. geology and exploration personnel, site inspections of the Kansoko decline portal and box-cut and the Kakula decline and box-cut, sites for mining, plant and infrastructure, discussions with other QPs and review of the existing infrastructure and facilities in the local area across the project.

Dr. Harry Parker visited the Kamoa-Kakula Project from 1 to 3 May 2009, from 27 to 30 April 2010, from 12 to14 November 2012, and from 17 to 19 January 2017. The site visits included presentations by Ivanhoe and African Mining Consultants' staff, inspection of core and surface outcrops, viewing drill platforms and sample cutting and logging facilities, and discussions of geology and mineralisation interpretations with Ivanhoe's staff. On his January 2017 visit, Dr. Parker checked drillhole locations, inspected drill core, and collected witness samples from the Kakula deposit.

Mr. Gordon Seibel visited the Project from 9 to 10 February 2011, from 5 to 8 November 2011, from 12 to 14 November 2012, and from 18 to 22 January 2016. During the site visits, Mr. Seibel inspected drill core, reviewed drill collar locations of new drilling in the field, took independent witness core samples, inspected the on-site sample preparation facility, and observed the sampling methodology and security measures from drill stem to laboratory pickup. The site visits also included discussions of geology and mineralisation interpretations with Ivanhoe's staff, focusing on deposit strike, dip, and faulting geometries. On his January 2016 visit, Mr. Seibel checked drillhole locations at Kakula, collected witness samples, and inspected drill core from the Kakula area.

Mr. William Joughin visited the site from 10 July 2017 to 13 July 2017 to review the geotechnical core logging and to inspect the ground conditions and support in the Kansoko decline and Kakula box-cut during construction. The site has been visited by personnel from SRK Consulting each of whom prepared a report on the site visit. The visits were undertaken on the dates as shown in Table 2.1.

Mr. Jon Treen visited the Kamoa Project site from 31 October to 1 November 2013 and the Kamoa and Kakula sites from 6 August 2018 to 8 August 2018. During the first visit, Mr. Treen inspected drill core, reviewed the drill core process, and inspected drills in operation on the site. Further inspection on the site of diamond drillhole collar locations, portal location, and tailings locations occurred. During the second visit, Mr. Treen inspected both declines at Kamoa and at Kakula. He also inspected drill core and the geotechnical core for the first ventilation riaise. Both visits included briefings from the Ivanhoe geological site management and project engineers. Mr. Treen also visited the KGHM operations in Poland to review the controlled convergence room-and-pillar mining method. The visit, from 26 to 28 January 2016, involved reviews of Lubin Mine, Runda Mine, and a review with their technology group Cuprum.

Mr Marius Phillips and Mr Gregory Ruiter from DRA Global have not visited the site, however the following DRA Global personnel visited the Kakula site:

- Mr. Thys de Beer, Project Manager for Kamoa-Kakula project employed by DRA Global, visited the site from 14 May 2018 to 18 May 2018. The site visit by Mr. de Beer included briefings from Ivanhoe geology personnel, site inspections of the Kakula decline portal and box-cut, potential areas for mining, plant and infrastructure, and a review of the existing infrastructure and facilities in the local area around the Kakula site.
- Mr. Wayne Venter, Project Engineer for Kamoa-Kakula project employed by DRA Global, has visited the Kakula site on the following occasions: 13 to 15 March 2018, 6 and 7 August 2018 and 12 to 15 November 2018. The site visit by Mr. Venter included a briefing from Ivanhoe mining and engineering personnel and site inspections of the Kakula decline portal and box-cut, potential areas for mining infrastructure at he Kakula site.
- Mr. Alwyn Scholtz, Senior Mining Engineer (Pr.Eng) for Kamoa-Kakula project employed by DRA Global, has visited the Kakula site from 9 July 2018 to 12 July 2018 and between 6 August 2018 to 7 August 2018. The site visit by Mr. Scholtz included site inspections of the Kakula decline portal and box-cut, as well as gaining an understanding of the current cost control systems in place.
- Mr. Danie Oosthuizen, Senior Civil Project Engineer for Kamoa-Kakula project employed by DRA Global, has visited the Kakula site on the following occasions: 10 December 2018 to 13 December 2018 and 25 June 2018 to 29 June 2018. The site visit by Mr. Oosthuizen included site inspections of the existing infrastructure, potential areas for new infrastructure at the Kakula site, discussions and meetings with Earthworks, Civils and SMPP contractors, airport road, discussing Infrastructure design philosophies and drawing reviews with site team.

Each of the above team members has provided briefings on their visits and conditions at the project to the two QPs.

Table 2.1 SRK Site Visits

Person	Dates	Purpose
Jarek Jakubec	27 April to 1 May 2010	Initial project geotechnical review.
Wayne Barnett	21 to 25 July 2010	Review progress in geotechnical characterisation and field work recommended by SRK in March 2010; and formulate an opinion on the structural deformation of the deposit and how it could impact the geotechnical characterisation of the deposit.
Ryan Campbell and Ross Greenwood	22 to 27 June 2011	Undertake QA/QC on current geotechnical logging practices. Alan Naismith and SRK Lubumbashi representatives were also on this visit.
Ross Greenwood and Desiré Tshibanda	5 to 12 August 2011	Geotechnical logging QA/QC.
Wayne Barnett	12 to 17 August 2011	Review the structural geology model development; review and update based on new drill core and orientated core measurements.
Ross Greenwood	12 to 19 February 2012	Geotechnical data collection QA/QC.
Wayne Barnett	13 to 17 June 2012	Carry out additional drill core observations and review the structural logging protocol in order to prepare the structural model to be derived for the Prefeasibility geotechnical study.
Desmond Mossop	18 to 20 November 2014	Geotechical Review of the Kansoko Box-cut, Portals and Decline Ground Control.
Shaun Murphy	July 2015	Geotechnical Review of the Kansoko Decline Ground Support review. Recommendations.
Rory Bush	25 July to 01 August 2016	Quality control. Decline Ground Support Recommendations.
Rory Bush	11 to 21 November 2016	Quality control. Geotechnical logging for the Kakula Decline Ground Support Recommendations.
William Joughin and Denisha Sewnun	10 to 13 July 2017	Geotechnical Review of the Kansoko Decline and Kakula box-cut Ground Support. Recommendations. Quality control.

2.5 Effective Date

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

- Effective date of the Report: 18 March 2019.
- Date of the database closure Kamoa Mineral Resource estimate: 23 November 2015.
- Date of the database closure Kakula Mineral Resource estimate: 26 January 2018.
- Date of drill information from the ongoing drill programme at Kakula: 1 March 2019.
- Date of drill information from the 2016–2017 drill programme at Kamoa and date of updated copy of the database: 27 November 2017. Information available from this data supply was used to validate the Kamoa geological model.
- Date of Mineral Resource estimate for Kamoa: 27 November 2017.
- Date of Mineral Resource estimate for Kakula: 13 April 2018.
- Date of Mineral Resource estimate for Kakula West: 10 November 2018.
- Date of the Mineral Reserve estimate for Kamoa; 1 February 2019.
- Date of the Mineral Reserve estimate for Kakula; 1 February 2019.
- Date of the supply of legal information supporting mineral tenure: 23 March 2018.

2.6 Information Sources and References

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

3 RELIANCE ON OTHER EXPERTS

The QPs, as authors of Kamoa-Kakula IDP19, 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.

3.1 Mineral Tenure

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

- Kamoa Copper SA: Report on the Kamoa-Kakula Project Property Description and Location, March 2019..
- Emery Mukendi Wafwana & Associates, SCP., 2016: Validity of (i) The Exploration Permits relating to The Mining Project of Kamoa; (ii) The Kamoa Exploitation Permits; (iii) The transfer of 45 of rest of The Kamoa Exploration Permits of Kamoa Copper SA to Ivanhoe Mines Exploration DRC SARL, addressed to Ivanhoe Mines Ltd.
- Andre–Dumont, H., 2013: Democratic Republic of the Congo: Report prepared by McGuireWoods LLP in Bourassa M.; and Turner, J., 2013 (eds): Mining in 31 jurisdictions worldwide 2013, Mining 2013, Getting the Deal Through, posted to http://www.mcguirewoods.com/news-resources/publications/international/miningdrcongo.pdf.
- Ivanhoe Mines DRC SARL, 2017, DRC Mining Code Review and Ministerial Decrees: Unpublished internal email prepared by Corporate Affairs Ivanhoe Mines DRC SARL, 28 June 2017.

This information was used in Section 4.3 of the Report and Section 14.13 for assessment of reasonable prospects of eventual economic extraction.

The QPs have also fully relied upon, and disclaim responsibility for, information supplied by Kamoa Copper SA for information relating to mineral tenure, ownership of the Project area, underlying property agreements, and permits through the following document:

- Kamoa Copper SA: Report on the Kamoa-Kakula Project Property Description and Location, March 2019.

This information was used in Section 4 of the Report, and Section 14.13 for assessment of reasonable prospects of eventual economic extraction.

3.2 Surface Rights

The QPs have fully relied upon, and disclaim responsibility for, information supplied by Kamoa Copper SA for information relating to payment of land and surface rights taxes and payment due dates for 2009–2017 through the following document:

- Kamoa Copper SA: Report on the Kamoa-Kakula Project Property Description and Location, January 2019.

This information was used in Section 4 of the Report, and Section 14.13 for assessment of reasonable prospects of eventual economic extraction.

3.3 Environmental and Work Programme Permitting

The QPs have obtained information regarding the environmental and work programme permitting status of the Project through opinions and data supplied by experts retained by Ivanhoe, and from information supplied by Ivanhoe staff. The QPs have fully relied upon, and disclaim responsibility for, information derived from such experts through the following documents:

- Kamoa Copper SA: Report on the Kamoa-Kakula Project Property Description and Location, January 2019.
- Kamoa Copper SA: Kamoa-Kakula Environmental and Social Report, March 2018.
- African Mining Consultants, 2009: Greater Kamoa Project, The Democratic Republic of the Congo, Environmental Impact Assessment Scoping Study: Unpublished report prepared by African Mining Consultants for African Minerals (Barbados) Ltd., Sprl, dated June 2009.
- Environmental Impact Study, by African Mining Consultants, dated April 2011, representing the original Environmental Impact Study approved by DRC Government.
- Environmental Social and Health gap analysis, by Golder dated March 2012: Report No. P1613890, containing the Environmental Social and Health gap analysis to assist in compiling the Environmental and Work Programme – Permitting.
- Kamoa Stakeholder Engagement Plan by Golder, dated September 2012: Report No. 11613890-11388-2 containing the Stakeholder Engagement Plan for the permitting of project components.
- Environmental Social and Health Constraints, by Golder dated August 2012: Report No. 11613890-11594-4 - Environmental Social and Health Constrains and Design Criteria assisting in the permitting process.
- Kamoa Environmental Social and Health Impact Assessment Scoping Study (Draft) by Golder dated August 2013, containing the detailed scoping report for IFC ESHIA.
- Kamoa Environmental Impact Study Terms of Reference (Draft) by Golder, dated August 2013 which contains the Terms of Reference Report for DRC regulations as part of the permitting process.

This information was used in Section 20 of the Report and Section 14.13 for assessment of reasonable prospects of eventual economic extraction.

3.4 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:

- Kamoa Copper SA: Email Re: Updated Taxes and Royalties to OreWin, 11 January 2019.
- Kamoa Copper SA: Report on the Kamoa-Kakula Project Property Description and Location, January 2019.
- 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 the Congo, dated 01 March 2016.
- Kamoa Copper SA, 2017: Kamoa Copper Project: Unpublished letter prepared by Kamoa Copper SA for OreWin, 26 June 2017.

This information was used in Section 22 of the Report and Section 14.13 for assessment of reasonable prospects of eventual economic extraction.

4 PROPERTY DESCRIPTION AND LOCATION

The Kamoa-Kakula Project is situated in the Kolwezi District of Lualaba Province, DRC. The Kamoa-Kakula Project is located approximately 25 km west of the town of Kolwezi, and about 270 km west of regional centre of Lubumbashi (see Figure 4.1).

The Project is centred at approximate latitude 10°46'S and longitude 25°15'E. The Project location is shown in Figure 4.1.

Figure 4.1 Project Location Map

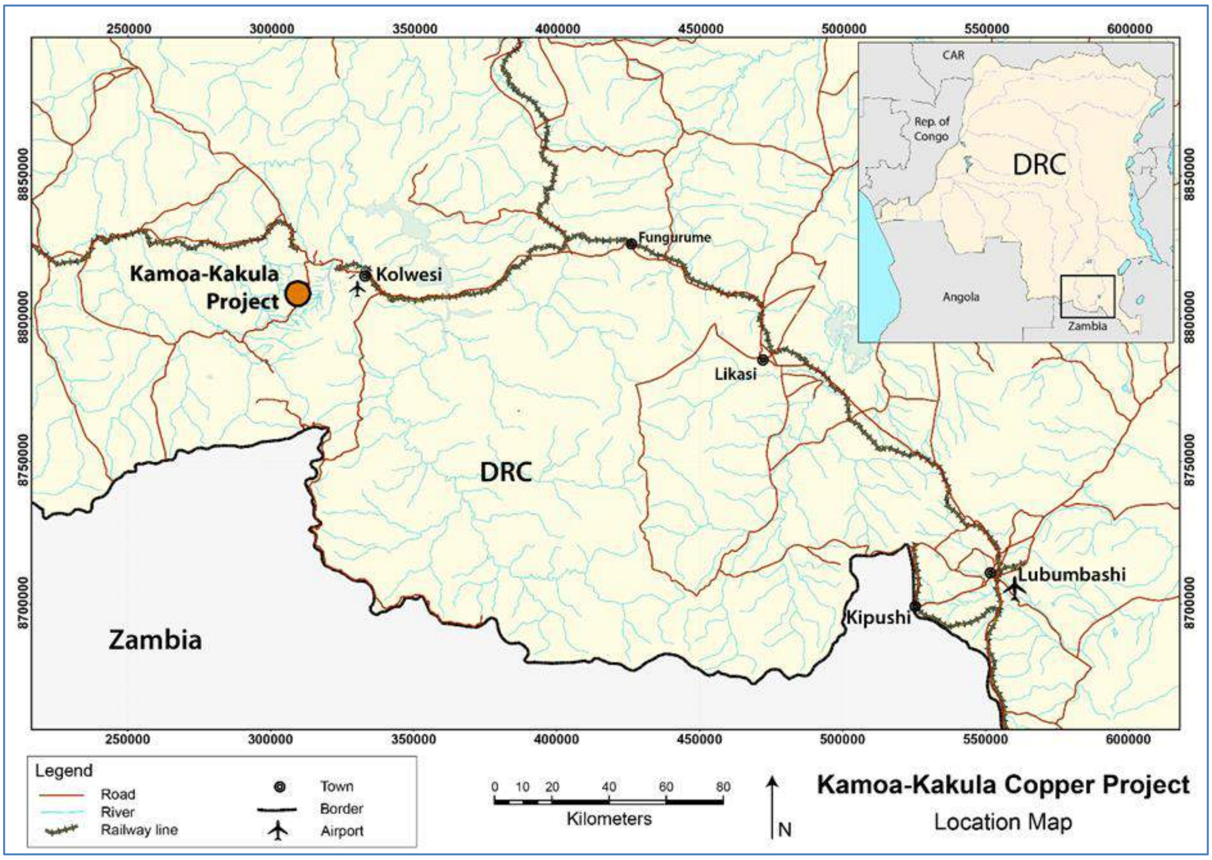


Figure by Ivanhoe, 2016.

4.1 Project Ownership

Ivanhoe owns a 49.5% share interest in Kamoa Holding Limited (Kamoa Holding), an Ivanhoe-Zijin subsidiary that presently owns 80% of the Kamoa-Kakula Project. Zijin owns a 49.5% share interest in Kamoa Holding, which it acquired from Ivanhoe in December 2015 for an aggregate cash consideration of US$412 million. The remaining 1% interest in Kamoa Holding is held by privately-owned Crystal River Global Limited. A 5%, non-dilutable interest in Kamoa Copper SA was transferred to the DRC following the shareholders' general meeting dated 11 September 2012, for no consideration, pursuant to the DRC Mining Code.

On 11 November 2016, Kamoa Holding and the DRC, represented by the DRC Minister of Mines and Minister of Portfolio, signed, in presence of Ivanhoe, Zijin Mining Group Co., Ltd. and Kamoa Copper, a share transfer agreement that transferred an additional 15% interest in the Kamoa-Kakula Project to the government of the DRC, increasing its total stake in the Project to 20%. As a result of the transaction, Ivanhoe and Zijin each hold an indirect 39.6% interest in the
Kamoa-Kakula Project while Crystal River Global Limited holds an indirect 0.8% interest and the DRC holds a direct 20% interest in the Kamoa-Kakula Project.

The share transfer agreement provides, without limitation, that:

- Kamoa Holding will transfer 300 Class A shares in the capital of Kamoa Copper SA—representing 15% of Kamoa Copper's share capital—to the DRC government, in consideration for a nominal cash payment and other guarantees from the DRC government summarised below. In addition, the DRC government owns 100 non-dilutable Class B shares, representing 5% of Kamoa Copper's share capital.
- The parties agreed that the 300 Class A shares shall be non-dilutable until the earlier of (i) five years after the date of the first commercial production and (ii) the date on which the DRC ceases to hold all of its 300 Class A shares.
- Kamoa Holding undertakes to provide all shareholder loans to Kamoa Copper SA and/or procure the project financing from third parties for the development of the Kamoa-Kakula Project.
- Kamoa Holding and the DRC government acknowledge that they shall not be entitled to any dividend on their shares in the share capital of Kamoa Copper SA before the repayment of 80% of all shareholder loans (which total US$819 million on 31 December 2018), and 100% of any financing of the project by third parties.
- The DRC government confirmed that the Kamoa-Kakula Project will be developed with the support of the government of DRC and of its Ministry of Mines by Kamoa Copper with the current and future shareholders of Kamoa Holding.
- The DRC government acknowledged and confirmed that all permits and mining rights currently held by Kamoa Copper in respect of the Kamoa-Kakula Project are at the date of the signature of the share transfer agreement valid and in good standing, without any defect and that Kamoa Copper's mining rights are not subject to any cancellation or to any litigation or dispute, whatsoever and recognised and guaranteed the peaceful enjoyment of its mining rights by Kamoa Copper.
- The DRC government confirmed and guaranteed that the Kamoa-Kakula Project will not be subject to any taxes or duties other than those legally required by the applicable statutory and regulatory provisions.
- The DRC government acknowledged and agreed that the interests on the shareholders' loan that was the subject of the technical opinion from the Department of Mines dated 13 November 2015 will be compliant with the terms approved by this opinion.
- At Kamoa Copper's request and subject to the satisfaction of the applicable conditions, the DRC State shall provide its assistance to Kamoa Copper, its affiliates and subcontractors for the purpose of obtaining the advantages contemplated by the DRC government's special law—No.14/005 dated 11 February 2014, determining the tax, customs, parafiscal tax, non-tax revenues and currency exchange regime applicable to collaboration agreements and cooperation projects.

- Kamoa Holding will have a preference right, and right of first refusal on any proposed sale, transfer or any, direct or indirect sale, transfer or other disposal by the DRC government of all or part of its 300 Class A shares in favour of a third party, in accordance with Article 13 of the articles of association of Kamoa Copper, the share transfer agreement clarifying the amendments of this provision to be adopted.
- The share transfer agreement will be governed by and construed in accordance with the laws of the DRC. Any dispute will be subject to binding arbitration, conducted in the French language, in Paris, France, in full accordance with the Convention on the Settlement of Investment Disputes between States and Nationals of Other States. An arbitral decision will be subject to enforcement under the New York Convention of 1958, to which the DRC government is a contracting party.

4.2 Property and Title in the Democratic Republic of Congo

4.2.1 Introduction

A summary of the mining history of the Katangan region is presented below, and is adapted from André-Dumont (2013), and from Law No.007/2002 dated 11 July 2002 on the Mining Code (herafter referred to as the "2002 Mining Code"), as amended and completed by Law No.18/001 dated 09 March 2018 (Mining Code).

The DRC contains a number of world-class Mineral Resources, including copper, cobalt, diamonds, and gold. Significant deposits of zinc, germanium, tin, tungsten, columbium tantalum (coltan), and uranium are also present.

The DRC has a long base-metal mining history, commencing with the formation of the Union *Minière* du Haut Katanga in 1906 and first industrial production of copper in 1911, from l'Etoile (Ruashi), a very rich copper oxide deposit located a few kilometres from Lubumbashi. Just prior to 1960, the DRC was the world's fourth-largest producer of copper and supplied 55% of the world's cobalt from deposits in Katanga. Following independence from Belgium in 1960, production gradually decreased due to a combination of factors that included political unrest, political and social environments within the country, declining investment in infrastructure, and lack of capital (Goossens, 2009).

In 1967, the DRC (then called Zaire) government nationalised private enterprise, creating the state-owned mining company *La Générale des Carrières et des Mines* (Gécamines). Despite controlling rich mineral deposits, the state company became unprofitable over time (Goossens, 2009). There followed, through war and disinvestment, a further destruction of general transport, energy, and telecommunications infrastructure.

A number of mineral concessions were granted by the DRC government from 1997 to 2001 to companies that wished to enter joint ventures with Gécamines. During 2007, following the first democratic elections in decades, the government of the DRC announced an initiative to review the mining agreements granted between 1997 and 2006 for Gécamines properties. This review did not affect the Kamoa-Kakula Project.

4.2.2 Mineral Property 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.

4.2.3 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 Kamoa Copper.

  Kamoa Copper and Kamoa Holding as well as the owner of the shares of Kamoa Holding consider that in spite of the above mentioned amendment of the 2002 Mining Code and with regard to international law, they are entitled to the respect by DRC of the 10-year stability guarantee granted by Article 276 of the 2002 Mining Code, notably for all the rights attached to its mining rights, including the tax and customs regimes and in accordance DRC's commitments made in the above mentioned share transfer agreement dated of 11 November 2016.

  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 Kamoa Copper,.

  Kamoa Copper and Kamoa Holding as well as the owner of the shares of Kamoa Holding consider that in requiring the immediate implementation of the more stringent tax requirements adopted by Law No.18/001, DRC breaches the clear and explicit stability guarantee that DRC undertook to respect in accordance with the 2002 Mining Code and the above mentioned share transfer agreement and whose validity cannot be challenged.

  With regard to the current contrary interpretation of DRC, in spite of the requests made by Kamoa Copper to have the stability guarantee respected by DRC and all its administrations during the stabilised period, Kamoa Copper 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 Kamoa Copper. Kamoa Copper already received tax adjustments for lack of compliance with new requirements on environment and expatriate taxes that are, in its view, not applicable to Kamoa Copper. Kamoa Copper 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 Kamoa Copper 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 Kamoa Copper with regard to the stability guarantee it is entitled to are nevertheless applied by DRC administrations to Kamoa Copper, for instance in relation to environmental taxes, expatriate taxes or explosives. They will increase the Kamoa-Kakula 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, Kamoa Copper 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. Kamoa Copper 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 Kamoa-Kakula Project and could be considered as being contradictory, without limitation, with the stability guarantee to which Kamoa Copper 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.

Kamoa Copper 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, Kamoa Copper 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. Kamoa Copper 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. Kamoa Copper 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. Kamoa Copper is still in the process of assessing whether or not they should apply to Kamoa Copper. Subject to further analysis and verification and to contrary interpretation from the DRC government authorities, Kamoa Copper's preliminary view is that those new requirements should not apply to Kamoa Copper.

Kamoa Copper, Kamoa Holding and the owner of the shares of Kamoa Holding consider that Kamoa Copper 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 Kamoa Copper 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.

4.2.4 Exploration Permits

An exploration permit, as defined in the Mining Code, grants to its holder the exclusive right to perform, within the perimeter over which it is established and during its validity period, exploration works of mineral substances classified in mines for which the exploration permit was granted and associated substances if the holder applies for the extension of the exploration permit to these substances.

Under the Mining Code, exploration permits are granted for a term of five years, and are renewable once for the same term.

No individual exploration permit can exceed a surface area of 400 km^2. One person and its affiliated companies cannot hold more than 50 exploration permits in the DRC, and the total granted area for all exploration permits within the DRC may not exceed 20,000 km^2.

Although applications are not subject to technical or environmental review, the applicant is subject to a requirement to prove appropriate financial capacity.

Renewal application automatically requires a 50% ground relinquishment. In other respects, under the Mining Code the holder of an exploration permit is authorised to take samples of the mineral substances within the Perimeter indicated on the Exploration Permit for analysis or industrial assays in the laboratory or plant of holder's choice.

However, the holder of an exploration permit must file at the Department of Geology of the Ministry of Mines a control sample (*échantillon témoin*) of all sample or samples batches taken within the Perimeter covered by the title.

An exploration permit holder can convert part of the permit into an exploitation permit, or a small-scale exploitation permit, and still maintain the rights to explore on the remainder of the exploration permit, subject to conditions laid out in the Mining Code.

4.2.5 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.

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.

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”.

4.2.6 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. Kamoa Copper was granted with such an authorisation from the Governor of the Province on 23 July 2014.

Kamoa Copper 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.

4.2.7 Environmental Regulations

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

All exploration, mining and quarrying operations must have an approved environmental plan, and the holders of the right to conduct such operations are responsible for compliance with the rehabilitation requirements stipulated in the plan. When applying for an exploitation permit, a company must complete an ESIS to be filed, together with the ESMP to be approved by the relevant authorities.

On approval, the applicant must provide a financial guarantee for rehabilitation. The security can be provided by means of a bank guarantee. Funds posted as security 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. Kamoa Copper complied with its obligation in this respect in accordance with the installments set out in the approved updated environmental impact study of the Project until 2017 and is in the process of finalizing the securing and filing of the financial guarantee required for 2018 and 2019. It is expected that it will be done in the course of the 1st semester of 2019.

Exploration Permit

Each exploration permit in the DRC requires a mitigation and rehabilitation plan (PAR in French acronym). The PAR sets out the type of exploration activity in the area and describes what measures will be carried out to ensure impacts are minimised and any significant damage is repaired.

The holder of a mining right submitted to the PAR must revise this initially approved plan:

- When the changes in the mining activities justify an amendment of the PAR.
- When a control and/or monitoring report demonstrates that the mitigation and rehabilitation measures planned in its PAR are no longer adapted and that there is a significant risk for the environment.

Exploitation Permit

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

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

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

The Mining Regulations also require an environmental audit every two-year period as from the date of approval of the initial ESIS. The report of the last bi-annual environmental audit concerning Exploitation Permits No. 12873, 13025, and 13026 was thus filed on 07 June 2018.

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

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

4.2.8 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.

Gross commercial value is determined by a coefficient depending on the nature of the product, which is 95% of total value for blister copper (91-98% Cu content) and 65% for copper concentrate (31-60% Cu content).

At the date of this Report, the copper concentrate that Kamoa Copper 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 Kamoa Copper holds mining rights that were valid when Law No.18/001 entered into force, Kamoa Copper, Kamoa Holding and the owner of the shares of Kamoa Holding consider that Kamoa Copper 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.

Kamoa Copper nevertheless notes the contrary interpretation from DRC administrations on similar issues and the opinion from Kamoa Copper, Kamoa Holding and the owner of the shares in Kamoa Holding is that in the event DRC would impose Kamoa Copper 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 Kamoa Copper is entitled to.

4.3 Mineral Tenure

The Kamoa-Kakula Project consists of the Kamoa exploitation permits (Exploitation Permits No. 12873, 13025, and 13026 which cover an area of approximately 39,316 hectares) and one exploration permit (Exploration Permit No. 703 which covers an area of approximately 1,260 hectares). A mineral tenure summary table is provided in Table 4.1 and the mineral tenure locations are as indicated in Figure 4.2. The Exploitation Permits were surveyed and boundary marked together with the Cadastre Minier. Exploration Permits are delineated by latitude/longitude co-ordinates and do not require survey.

Table 4.1 Permit Summary Table

Exploitation Permit (PE) No.	Grant Date	Expiry Date	Mineral/Metal Rights Granted	Number Cadastral Squares (Quadrangles)	Area (ha)*
12873	20 Aug 2012	19 August 2042	Silver, Bismuth, Cadmium, Cobalt, Copper, Iron, Germanium, Nickel, Gold, Palladium, Platinum, Lead, Rhenium, Sulphur and Zinc.	62	5,207.67
13025	20 Aug 2012	19 August 2042	Silver, Bismuth, Cadmium, Cobalt, Copper, Iron, Germanium, Nickel, Gold, Palladium, Platinum, Lead, Rhenium, Sulphur and Zinc.	204	17,135.69
13026	20 Aug 2012	19 August 2042	Silver, Bismuth, Cadmium, Cobalt, Copper, Iron, Germanium, Nickel, Gold, Palladium, Platinum, Lead, Rhenium, Sulphur and Zinc.	202	16,972.25
Sub Total					**1,260.17**
703 (Exploration Permit)	11 Nov 2003	10 May 2020	Base, Precious, Platinum Group Metals, Pegmatite Minerals, Diamonds and Gemstones	15	1,260.17
Sub Total					**1,260.17**
Total					**40,575.78**

*The above-mentioned areas are calculated in the GIS environment.

Figure 4.2 Project Tenure Plan

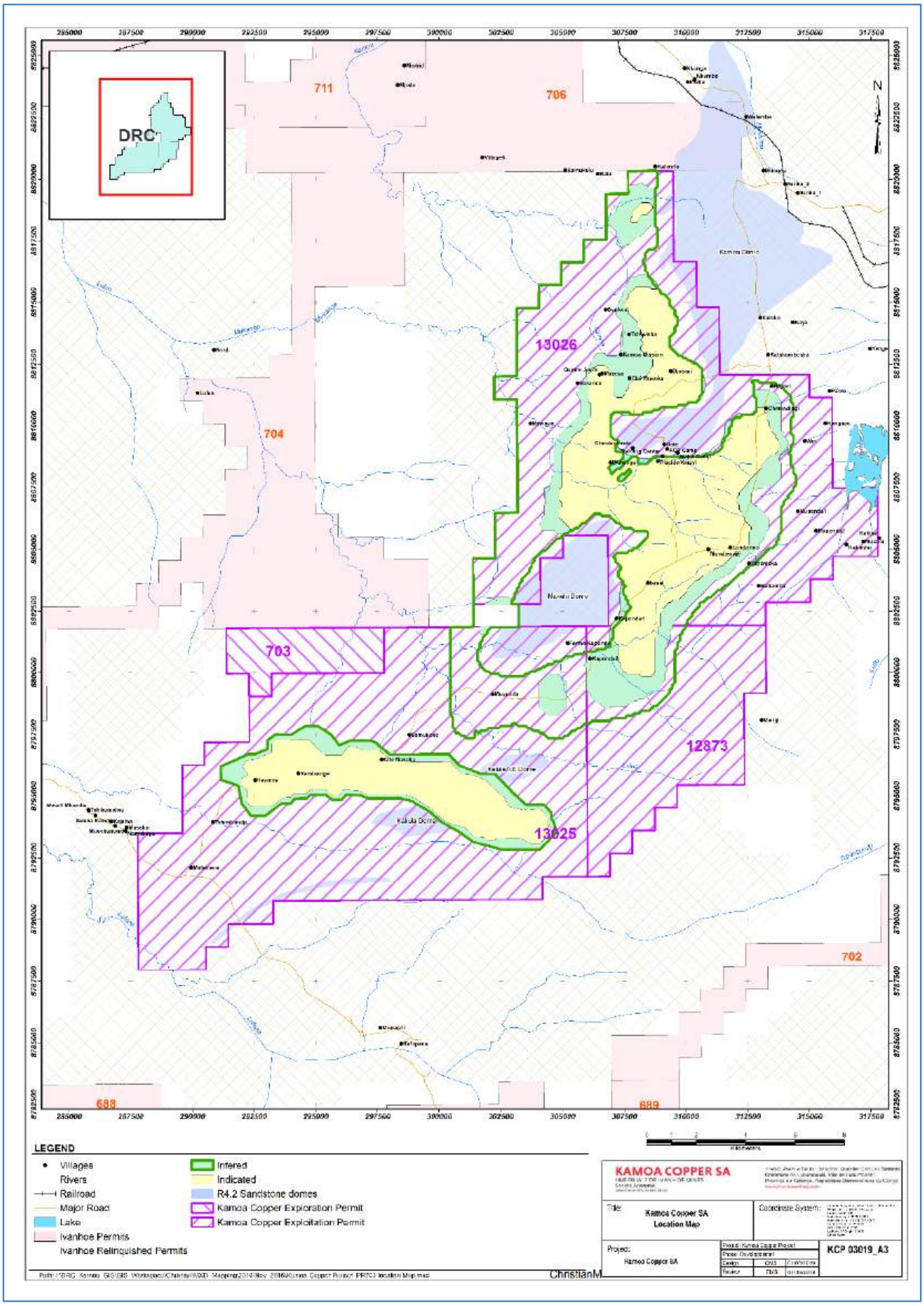


Figure by Kamoa Copper SA, 2019.

Ivanhoe advised the QPs that Ivanhoe had pro-rata paid the required annual superficiary rights for the Exploitation Permits to the DRC Government, as this pre-payment was a pre-condition of grant of the permits. The annual superficiary rights are due by 31 March of each year; Tax on the area of mining concessions is due by 1 February of each year. Ivanhoe advised the QPs that the required payments for 2018 were made for the three above-mentioned Exploitation Permits and for Exploration Permit 703, and that Kamoa Copper SA has paid the required fees at the date of this report.

Ivanhoe is also actively exploring in other areas of the DRC, with exploration permit tenure holdings which are at a grass-roots exploration stage.

4.4 Surface Rights

At the effective date of this Report, Kamoa Copper SA holds no surface rights in the Project area but is authorised to occupy the parcels of land required for its activities. Investigations with local administrations should be performed to clarify whether or not there are any holder of surface rights enforceable against third parties within the area of planned infrastructure. Land access for the exploration programmes completed to date has typically been negotiated without problems. Where compensation has been required for exploration activities, compensation has followed DRC laws and regulations in all cases. The surface rights for the whole surface covered by the mining rights belongs to the DRC State. Kamoa has completed a process of compensation to communities and individual farmers for the loss of land and for fields inside the 7 km^2 required for the Kansoko mine as required by the DRC law to enable the company to occupy this land. A similar process was performed for Kakula, including the resettlement of 45 households surveyed in the Kakula footprint. The land replacement is in progress. This is a process which will go over two years. The mine area was fenced off, Kamoa Copper is in the process of creating a pathway for bikes as a deviation road so that people cultivating in the south area beyond the fence can easily access their fields. Kamoa Copper could consider in the future applying for prohibition areas (zones d'interdiction) where the activities and/or circulation of third parties will be prohibited for the areas required for the Kansoko and Kakula surface infrastructure that give the company the full legal right to occupy the relevant area and prevent any other parties occupying or entering the area.

4.5 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. Kamoa Copper has successfully received such exoneration in the past and expects to receive such exoneration for most imports for project construction.

4.6 Property Agreements

There are no agreements in place that are relevant to the Technical Report.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

5.1.1 Air

The city of Lubumbashi in the DRC, located 290 km east of the Kamoa-Kakula Project, can be accessed by an international airfield. Alternatively, the international airport at the Zambian city of Ndola, 200 km south-east of Lubumbashi, can be used.

The closest major township to the Project is Kolwezi, 25 km to the east. There are regular flights from Lubumbashi to Kolwezi, with the flying time being approximately 45 minutes.

5.1.2 Road

Kolwezi is connected to Lubumbashi and Ndola by road. Travel time by car from Kolwezi to Lubumbashi is currently four hours on a tarred road that has recently been refurbished and is in reasonable condition.

Access to the Project area from Kolwezi is via gravel roads to the villages of Kasekelesa and Musokantanda. Some of the gravel road network throughout the Project has been upgraded by Ivanhoe to provide drill and logistical access.

A 10 km road from the Kansoko mine site to Kakula has been constructed to facilitate access for drill rigs and construction equipment during the rainy season.

5.1.3 Rail

Until 2012, the rail line of approximately 740 km between Ndola (border with DRC) and the Livingstone (border with Zimbabwe) was managed under concession by RSZ (Railway System of Zambia). This concession was revoked in September 2012 and is currently run under management of the Zambian government.

The operation of the 470 km section between Bulawayo and Victoria Falls (Livingstone) on the Zambia border is carried out by the National Railways of Zimbabwe (NRZ) with NLPI Logistics (NLL) responsible for the financing and marketing of the line, per the agreement between NLL and NRZ. The 350 km railway line from Beitbridge (the border post between South Africa and Zimbabwe) to Bulawayo (the most industrialised city in Zimbabwe) was built in record time, with the construction phase lasting only 18 months. Implemented in Zimbabwe on a Build-Operate-Transfer basis by Beitbridge Bulawayo Railway BBR, it is now run by the NRZ.

Transnet Freight Rail (TFR) is the rail operator of the freight rail network in South Africa, and Transnet owns the assets. The railway system has sections running at world class standards, maintaining high volumes over long distances. TFR has an investment plan based on a forecast volume increase and new rail customers, which includes an upgrade of the line and a purchase of additional rolling stock to manage increased demand. TFR is a South African government-owned company.

A large port such as Durban exports bulk, break-bulk and containers fed by block trains of 100 or more wagons (railcars).

The condition of, and access to, the current rail infrastructure in the DRC makes rail a less viable option for inbound Project logistics.

5.2 Climate

The climate in the area follows a distinct pattern of wet and dry seasons. Rainfall of approximately 1,225 mm is experienced annually in the region with the majority of rainfall events occurring during the period of October through to March (the wet season), with peak precipitation being experienced between December to February. The dry season occurs from April to September. The average air temperature remains very similar throughout the year, averaging approximately 22°C. The average annual temperatures in the vicinity of the Kamoa deposit vary between 16°C and 28°C, with the average being 20.6°C. Winds at the Kamoa-Kakula Project are expected to originate from the east-south-east 20% of the time and south-east 14% of the time. Wind speeds are moderate to strong, with a low percentage (11.25%) of calm conditions (<1 m/s).

5.3 Local Resource and Infrastructure

The Project is currently isolated from public infrastructure. Infrastructure on-site is currently limited to support for the exploration activities. Exploitation of the Kamoa and Kakula deposits will require building a greenfields project with attendant infrastructure. Processing infrastructure exists in the Kolwezi mining district, but it is unknown whether this could be utilised by the Project.

5.4 Power

The bulk power supply is sourced from SNEL (La Société Nationale d’Electricité), the national power utility of the Democratic Republic of Congo. Capacity from the national grid is reserved through a partnership project between SNEL and Ivanhoe Mines Energy DRC, a subsidiary of Kamoa Holding Ltd. The partnership project is the rehabilitation of six turbine generators at the Mwadingusha hydropower plant in south east DRC. The Mwadingusha Dam, impounds the Lufira River creating the Mwadingusha Reservoir, and the facility was originally commissioned in 1930. Once completed, the fully upgrade and modernization project is expected to restore Mwadingusha to its installed capacity of about 71 MW. After completion and hand over, HPP Mwandingusha will supply electrical energy to the Congo National Grid as well as to the copper mining activities at the Kamoa-Kakula project by Ivanhoe mines. The project is funded by Ivanhoe Mines Energy DRC on a loan agreement with SNEL that will be repaid on a 40% discounted consumption charge.

5.5 Physiography

The Project area is at the edge of a north–north-east to south–south-west trending ridge which is incised by numerous streams and rivers. The elevation of the Project area ranges from 1,300 m to 1,540 m above sea level (amsl), with current exploration activities in areas of elevation from 1,450 m to 1,540 m above sea level (amsl). The local topography of the Project is affected by the drainage catchments of the Mukanga, Kamoa, and Lulua Rivers and the Kalundu, Kansoko, and Kabulo Streams.

The Project lies just north of the watershed separating the Zambezi and Congo drainage basins. Mukanga, Lwampeko, Kansoko, and Kamoa are the main streams in the Project area. These are the main sources of potable water for the local communities. Wetland areas in the general Project area include dambos (water-filled depressions), marshes, and wet plateau sands.

The Project is generally well vegetated with Central Zambezian Miombo woodland, characterised by broadleaf deciduous woodland and savannas interspersed with grassland, wetlands, and riparian forests. Grasslands on the Kalahari Sand plateau, together with riparian forests, are the most common vegetation type after Miombo woodland. Riparian forest dominates adjacent to watercourses.

There are no known migratory routes of endangered animal species within the Project area. Information gathered from interviews with local people indicates that the only protected species in the Project area are tortoises, which occur across the whole area. The partially protected felis serval (serval) is also found within the area. Poaching has severely diminished the numbers of larger mammals.

The most common vegetation disturbance is agriculture, and in particular the practice of slash-and-burn cultivation. There is currently little evidence of commercial logging, probably due to the poor road infrastructure. Woodland is only cleared or partially logged near villages where the need for agricultural land and firewood (charcoal) is greatest. No plant species threatened by extinction were found in the Project area during the surveys.

5.6 Comments on Section 5

The existing and planned access, infrastructure, availability of staff, the existing power, water, and communications facilities, the methods whereby goods could be transported to any proposed mine, and any planned modifications or supporting studies are reasonably well-established. There is sufficient area in the Project tenure to support construction of plant, mining and disposal infrastructure. The requirements to establish such infrastructure are reasonably well understood by Ivanhoe. It is expected that any future mining operations will be able to be conducted year-round.

6 HISTORY

During the period between 1971 and 1975, the Tenke Fungurume Consortium (consisting of Amoco, Charter, Mitsui, BRGM and L. Tempelsman, and operated as the Societé Internationale Des Mines du Zaire (SIMZ)), undertook grass-roots exploration over an area that extended south-west from Kolwezi toward the Zambian border. A helicopter-supported regional stream-sediment sampling programme was completed in 1971. No sample location information is available for any sampling that may have occurred within the confines of the current Project.

In 2003, Ivanhoe acquired a significant ground holding, including the permit areas that now comprise the Project. Work completed to date includes data compilation, acquisition of satellite imagery, geological mapping, stream sediment and soil geochemical sampling, an airborne geophysical survey that collected total field magnetic intensity, horizontal and longitudinal magnetic gradient, multi-channel radiometric, linear and barometric, altimetric and positional data, acquisition of whole-rock major and trace element data from selected intervals of mineralised zone and footwall sandstone in drillhole DKMC_DD019, and aircore, reverse circulation (RC) and core (DDC) drilling.

A first-time Mineral Resource estimate was prepared by Amec (now known as Amec Foster Wheeler) for the Kamoa deposit in 2009, and the estimate was updated in 2010, 2011, 2012, 2013, 2016, 2017, 2018 and has now been updated in 2019.

PEAs on the Kamoa deposit were prepared in 2012 (Peters et al., 2012), 2013 (Peters et al., 2013), 2016 (Peters et al., 2016) and 2017 (Peters et al., 2018).

The Kansoko Mine has a Mineral Reserve that was previously stated in the Kamoa 2016 Prefeasibility Study (Kamoa 2016 PFS). The base case described in the Kamoa 2016 PFS is the construction and operation of an underground mine, concentrator processing facilities, and associated infrastructure. The base case mining rate and concentrator feed capacity is 3 Mtpa. The production rate was increased to 6 Mtpa and mining methods changed for the Mineral Reserve update, in the Kamoa 2017 PFS. The Kamoa 2016 Resource Technical Report was filed in November 2016 that included a first-time resource estimate for the Kakula deposit. In January 2017 the Kakula 2016 PEA was filed. The Kakula 2016 PEA included an analysis of the Kakula deposit as a standalone operation and a combined operation that is made up of the separate operations at the Kansoko Mine and the Kakula Mine at the Kakula deposit.

The Kakula 2017 Resource Update was released in a Technical Report in June 2017, this was followed by the Kamoa-Kakula 2017 Development Plan which was filed in January 2018. The Kamoa-Kakula 2017 Development Plan included an update of the Kamoa Mineral Reserve and updates of the PEA on the Kakula Mineral Resource. The production rate assumption at each deposit has increased from 4 Mtpa to 6 Mtpa, and the total combined production rate has increased from 8 Mtpa to 12 Mtpa. The Mineral Reserves for the Kamoa 2017 PFS increased as a result of an increase in production rate through a change to the controlled convergence room-and-pillar mining method.

The previous Technical Report was the Kamoa-Kakula 2018 Resource Update with an effective date in March 2018, this included a restatement of the Kamoa-Kakula 2017 Development Plan which had been released in a Technical Report with an effective date in March 2018

7 GEOLOGICAL SETTING AND MINERALISATION

The discussion in this section has been prepared from published papers on regional geology as cited, particularly Selley et al (2018) and Kamoa-specific papers, particularly Schmandt et al (2013), Kennedy et al (2018) and Twite et al (2019), and is also based on discussions with, and presentations made by, Ivanhoe personnel (David Broughton, David Edwards, and George Gilchrist).

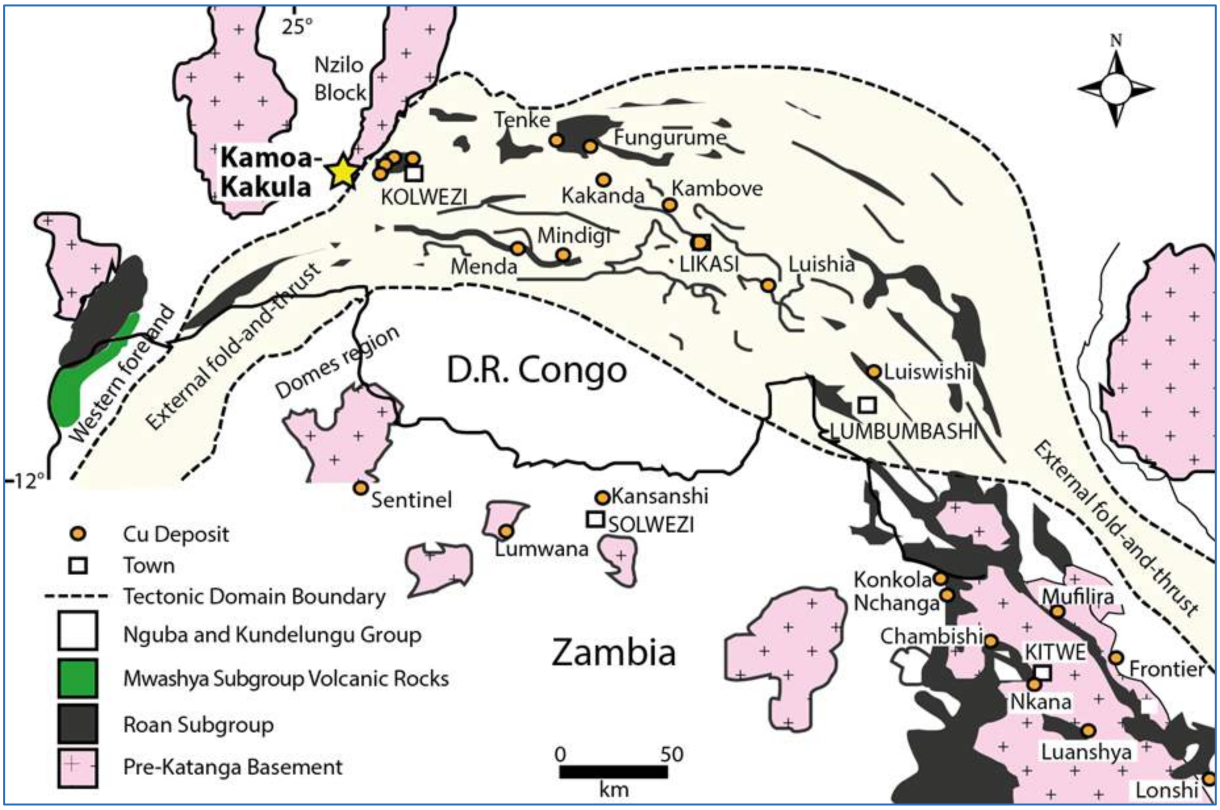
7.1 Regional Geology

The metallogenic province of the Central African Copperbelt is hosted in metasedimentary rocks of the Neoproterozoic Katanga Basin. 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. These units ("Ore Shale") contain the bulk of the deposits within the Copperbelt (the Kamoa deposit is, however, an exception to this). 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. The shape of the orogen is defined by a convex-northward array of folds and reverse faults (the Lufilian Arc), most clearly shown by the curvilinear outcrop patterns of Roan Group strata in the Katangan portion of the Copperbelt (Figure 7.1).

7.1.1 Stratigraphy and Bbasin Ddevelopment

The Katangan Basin overlies a composite basement made up of older, multiply-deformed and metamorphosed intrusions that are mostly of granitic affinity and supracrustal metavolcanic–sedimentary sequences. In Zambia, this basement is mainly Paleoproterozoic in age (2,100–1,900 Ma), whereas in the Kamoa region, only Mesoproterozoic basement (~1,100–1,300 Ma) is known.

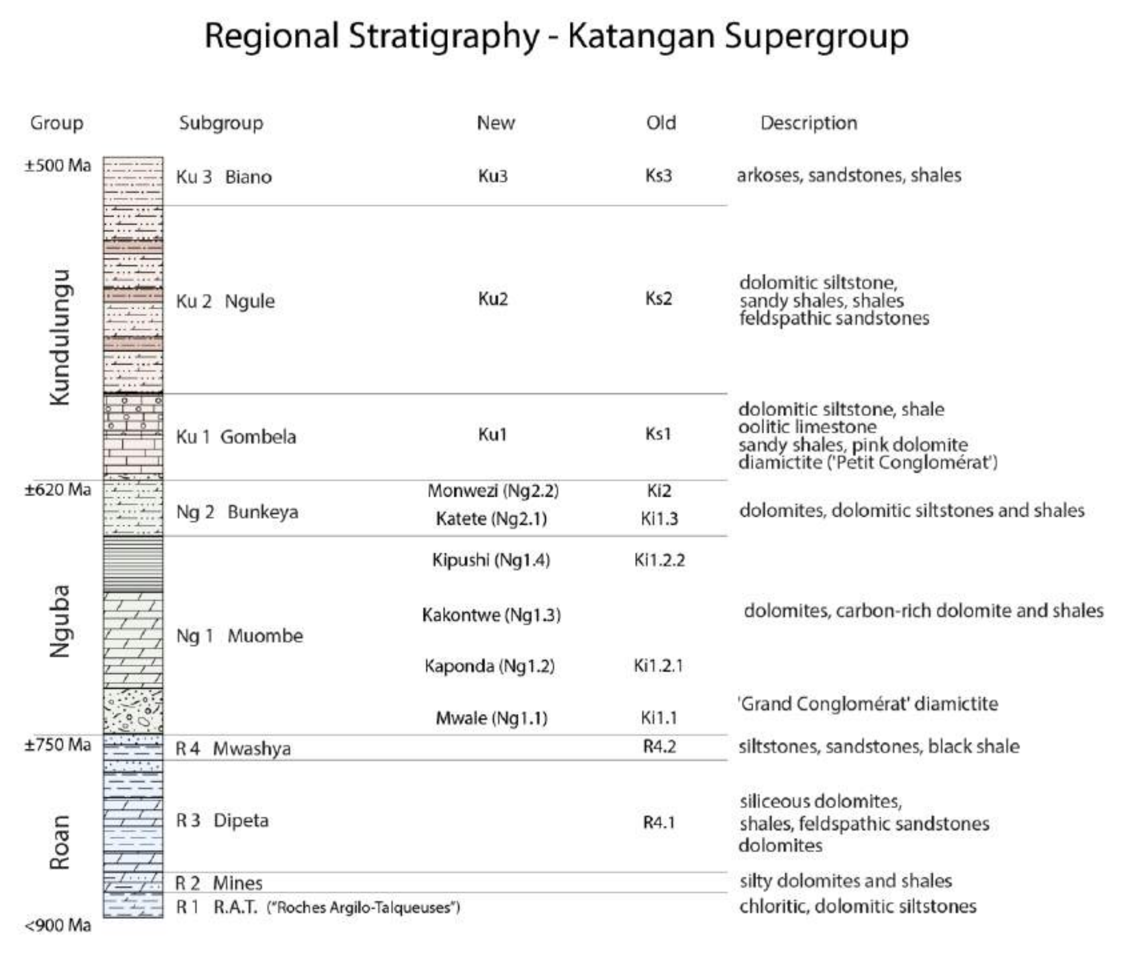
Figure 7.1 Geological Setting Central African Copperbelt



Source: Adapted from Schmandt et al (2013).

Nomenclature can be confusing for the 5 km to 10 km thick Katanga Supergroup. The DRC sector is currently subdivided into the Roan (R), N'Guba (Ng) and Kundulungu (Ku) Groups, (refer to Figure 7.2). The N'Guba and Kundulungu Groups were previously known as the Lower Kundelungu or Kundelungu Inferieur (Ki), and Upper Kundelungu or Kundelungu Superieur (Ks) Groups respectively. Geological and lithological descriptions in use at site, and thus in this Report, use the earlier nomenclature.

Figure 7.2 Stratigraphic Sequence, Katangan Copperbelt



Modified from Francois, 1973, 1974, 1987; Buffard, 1977, Batumike et al, 2007; Cailteux et al., 2007 and Heijilen et al. 2008

The metasedimentary rocks of the Lower Roan Group were deposited in an environment that was initially terrestrial in character but evolved to a marine character during a regional transgression. In the basal Roan Group, temporarily anoxic conditions in a lagoonal to mudflat environment prevailed, giving rise to intercalations of evaporite-bearing rocks in the siliclastic–carbonate successions.

The onset of the second rift phase was marked by an abrupt transition from platformal carbonate-facies rocks to largely siltstone-facies rocks of the Mwashya Subgroup (uppermost part of the Roan Group) and a distinctive, variably reducing glaciogenic diamictite, the Grand Conglomerat. During this phase of deposition, there was a progressive widening of the basin so that younger strata lap onto basement at the basin periphery (Selley et al., 2018).

The Grand Conglomerat is one of two recognised glaciogenic formations (the other being the Petit Conglomerat) within the Katangan. It is developed at the base of the Lower Kundulungu Group. It is widely developed throughout the Lufilian belt region and is capped by the Kakontwe Formation (Wendorff and Key, 2009). It is characterised by massive, matrix supported diamicties with clasts that vary from granules to boulders. The matrix is typically very mud or silt rich, with interbedded varved shale or siltstone layers. Dropstone clasts are often evident in these finely layered sequences (Binda and Van Eden, 1972). These features are considered evidence that the Grand Conglomerat is of glacial origin, or resedimentation of glacial sediments (Binda and Van Eden, 1972; Wendorff and Key, 2009).

7.1.2 Structural Framework

The Katangan basin was inverted during the Pan-African Lufilian orogeny, from approximately 580 Ma to 500 Ma. The Lufilian Arc can be divided into subregions, of which the Katangan (Congo) Copperbelt in the DRC belongs to an outer terrane of the arc, the External Fold and Thrust Belt. 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.

All of the Mines Subgroup copper (+/- cobalt) orebodies of the Katangan Copperbelt occur as megafragments (écailles) up to kilometres in size, within this megabreccia. The Kolwezi district comprises megafragments of the Mines Subgroup emplaced above the level of Ks2.1 strata (refer to Figure 7.3).

Kamoa occurs outside of this domain, with a far simpler structural configuration, similar in style to the southern Congolese and Zambian portions of the Copperbelt, and in sharp contrast to the complex strain patterns of the neighbouring Kolwezi district. The uppermost Roan and Kundulungu Group rocks are gently deformed, with open, upright folds trending east-north-east and west-north-west, forming dome-and-basin patterns, interpreted to be inherited from subbasin architecture associated with the Mwashya Subgroup rift stage (Selley et al., 2018).

7.1.3 Mineralisation

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, except in extreme basin marginal positions such as Kamoa, where primary salt was not deposited. Stratiform Cu ± Co ores occur at intrasalt (Congolese-type), subsalt (Zambian-type), and salt-marginal (Kamoa-type) positions (Selley et al., 2018). Mineralisation in the majority of the Katangan Copperbelt orebodies such as at Kolwezi and Tenke–Fungurume (Figure 7.1) is hosted in the Mines supergroup (R2). The mineralisation at Kamoa differs from these deposits in that it is located in the Grand Conglomerat unit (Ki1.1) at the base of the lower Kundelungu Group.

Mineralising fluids appear linked to residual evaporitic brines generated during deposition of the basin-wide salt-sheets, occupying large subsalt and intrasalt aquifers from ~800 Ma. This marks the earliest likely mineralising event, particularly in the Zambian-type stratiform Cu ± Co ores, which had continuous interaction with subsalt aquifers over long periods due to the static (nonhalokinetic) character of the salt in this region (Selley et al., 2018).

Whilst timing of mineralisation may be a focus of debate, the absolute age of mineralisation is a minor consideration in exploration for sediment-hosted Cu deposits as the distribution of deposits is fundamentally controlled by the chemical and permeability characteristics upon which ore fluids were superimposed (Selley et al., 2018).

Figure 7.3 Location of the Kamoa-Kakula Project in Relation to the Regional Geology of the Kamoa and Kolwezi Area

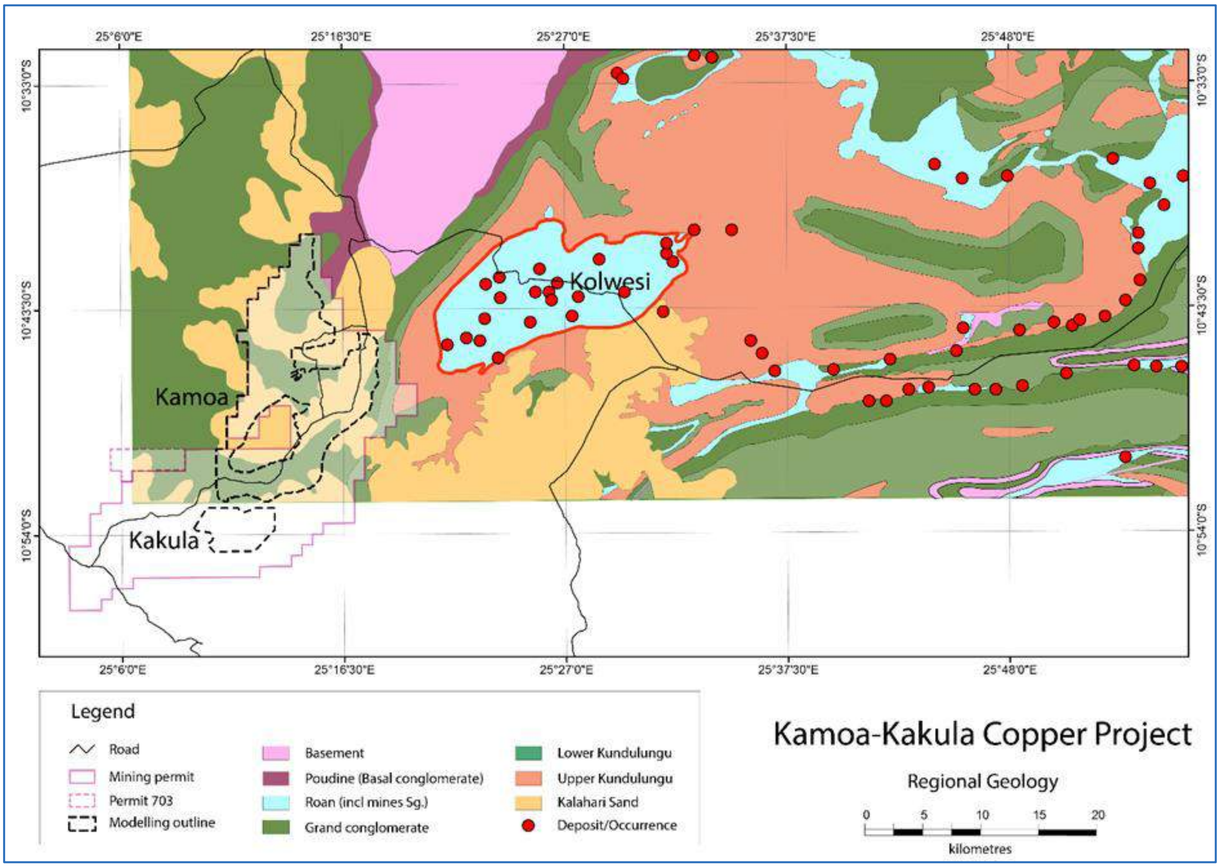


Figure provided by Ivanhoe, 2016.

7.2 Project Geology

The modelled Kamoa deposit is located in a broadly-folded terrane centred on the Kamoa and Makalu domes between the West Scarp Fault and Kansoko Trend. The Kakula deposit is located in a broadly folded terrane with the central portions of Kakula, and Kakula West, located on the top of the antiforms. The domes form erosional windows exposing the redox boundary between the underlying haematitic (oxidised) Roan sandstones, and the overlying carbonaceous and sulphidic (reduced) Grand Conglomerat diamictite (host to mineralisation). The outline of the domes used in the resource model are expanded to include portions of the Grand Conglomerat that have been leached of mineralisation. Unlike the tectonically-dismembered deposits of the Katangan Copperbelt, and the External Fold and Thrust Belt, the host rocks at Kamoa-Kakula are intact and relatively undisturbed.

Two primary trends are evident on the Project and are interpreted to be inherited from the underlying subbasin architecture. A first-order north-east-trending anticline and second-order east-north-east-trending synforms occur at Kamoa and project towards Kolwezi. Second-order west-north-west-trending synforms occur at Kakula, broadly conforming to the trend of the regionally- developed Monwezi Ffault zone of the central Congolese Copperbelt (Selley et al., 2018). Basin growth during deposition of the Grand Conglomerat is evident in a progressive thickening to the south-west.

For reference to different areas within the Kamoa deposit, the Project area was divided into 13 prospect areas that are referred to throughout this Report (refer to Figure 7.4). In the succeeding presentation in Sections 7 to 14, two forms of copper are recognised: total copper is designated as TCu; sulphuric acid soluble copper is designated as ASCu.

Figure 7.4 Prospect Areas Within the Combined Exploitation Permits

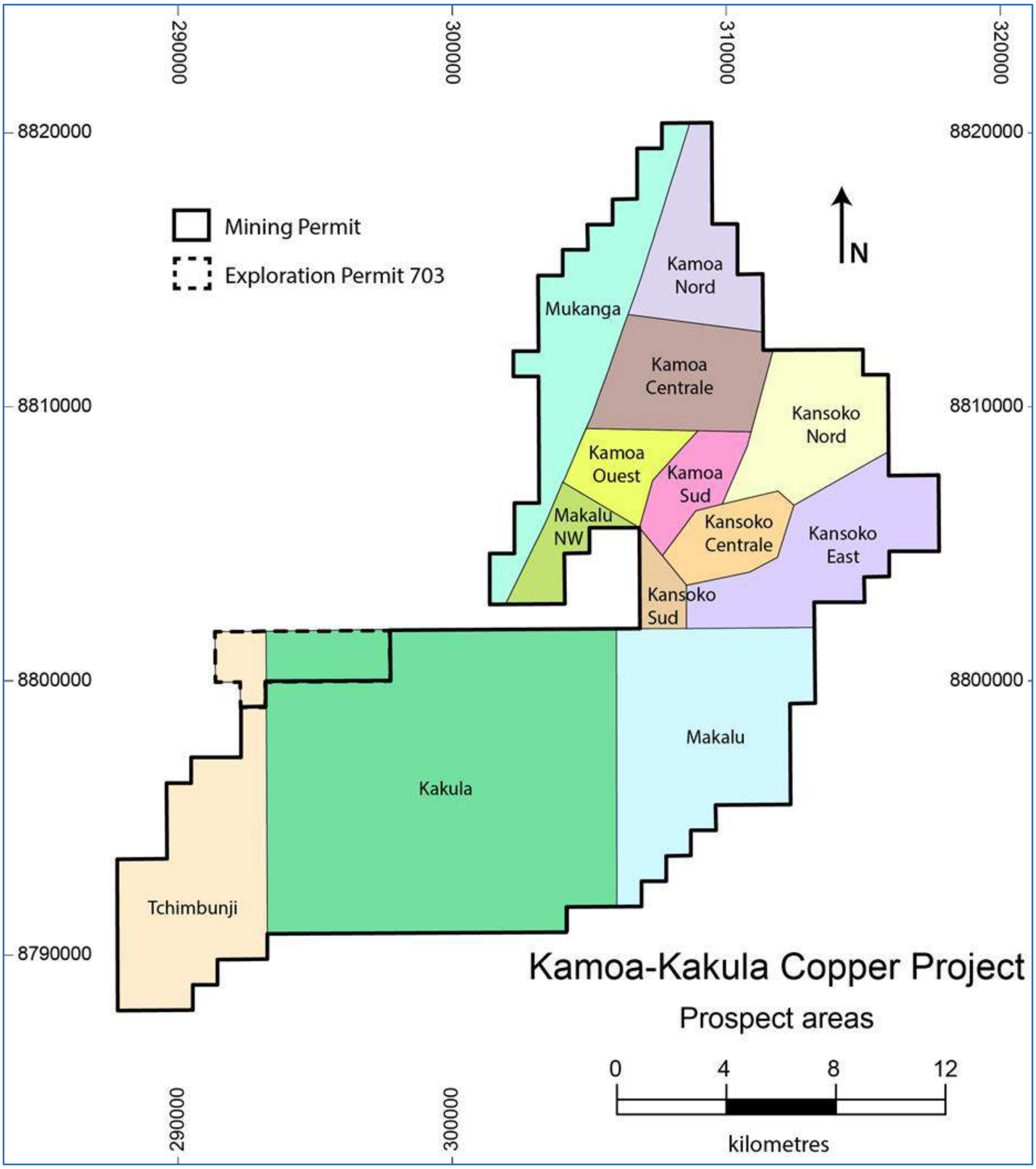


Figure provided by Ivanhoe, 2016.

7.3 Deposit Description

At Kamoa, haematite-bearing sandstone and siltstone of the Mwashya Subgroup (upper Roan Group) form the oxidised lower strata, and the pyritic rocks of the basal diamictite and inter-bedded siltstone-sandstones form the reduced host rock (Twite et al. 2018).

Although often associated with glacial origins, the diamictites of the Grand Conglomérat at Kamoa are interpreted as the deposits of cohesive debris flows, with the sandstone and silt-stone units the product of turbidity flows in a rapidly subsiding and evolving rift (Kennedy et al. 2018). The abundance of framboidal pyrite, which can only form under anoxic conditions, suggests there was little shallowing of the basin even with the substantial sedimentary input (Kennedy et al. 2018). This pyrite played a critical role in providing the reductant for copper sulphide mineralisation in diamictites and siltstone units at the base of the Grand Conglomérat (Schmandt et al. 2013).

7.3.1 Stratigraphic Sequence

Within the Project area, separate localised stratigraphy has been recognised for both the Kamoa and Kakula deposits, defining greater detail for the basal Ki1.1.1 diamictite of the Lower Kundulungu Group, (refer to Figure 7.5 and Figure 7.6 for Kamoa, and Figure 7.9 for Kakula). Sandstones of the Mwashya Subgroup of the Roan Group form the basal unit (R4.2) and are known from three drillholes on the northern and southern limits of the Makalu Dome, and one at Kakula, to have a thickness in the order of hundreds of metres.

At Kamoa, the R4.2 is overlain by a clast-rich diamictite (Ki1.1.1.1), identified by its percentage of clasts (20% to 35%), colour (maroon to light grey), sandy matrix, frequent matrix and replacement haematite alteration, and general lack of economic mineralisation. In turn, this is overlain by a clast-poor diamictite (Ki1.1.1.3), which is characterised by its percentage of clasts (<20% typically), an argillaceous to sandy matrix that is frequently chloritised, and its reduced nature, acting as the regional reductant in the Project area, refer to Figure 7.7. Mineralisation is typically concentrated along the basal contact of this diamictite, or in an intermediate siltstone (Ki1.1.1.2) at its base that is locally developed separating the two diamictites. This siltstone is typically massive to weakly bedded and can often be quite sandy, with bands of fine grit and reworked clasts. The Ki1.1.1.2 can frequently be a zone of intercalated siltstone, sandstone and diamictite, particularly to the south-west in the Makalu area where it more closely resembles the numerous siltstones developed at Kakula, or along north-west trending zones that may indicate the position of syn-sedimentary faults. Where intercalated layers are developed, mineralisation of the unit can be quite variable in response to the changes in the underlying lithologies, giving rise to complex grade profiles.

A regionally developed, finely-laminated, pyritic siltstone known as the Kamoa Pyritic Siltstone, or KPS (Ki1.1.2), is developed above the diamictite units at both Kamoa and Kakula. Sandy or gritty layers are developed within the siltstone; conglomerate layers are locally developed towards the base of the unit. Pyrite can range from fine to coarse-grained, but as shown in Figure 7.8, even where coarse grained, the pyrite still occurs concordant to the bedding planes. The basal contact of the KPS is marked by very finely layered varves(refer to Figure 7.8). Dropstones (also shown in Figure 7.8) can be seen to cause soft-sediment deformation. At Kamoa, the KPS can host mineralisation along the basal contact where the clast-poor (Ki1.1.1.3) diamictite is absent. The KPS is overlain by a thick sequence of diamictite with laterally discontinuous siltstone layers (Ki1.1.3).

The Ki1.1.4 is a regionally developed light, medium to greenish-grey bedded to laminated pyritic siltstone with intercalations of light grey sandstone and minor gritty pebbles. As with the KPS, the fine laminations highlight soft sedimentary deformation and syn-sedimentary folds. Reworked textures are also commonly observed within this unit (Twite, 2016). The Ki1.1.4 is overlain by a thick (>300 m) unit of light greensigh grey, clast poor diamictite (Ki1.1.5). A relatively thick (average 24 m), distinctive, cross-bedded sandstone separates the Ki1.1.5 from the overlying Ki1.1.6 diamictite, which is similar in character to the Ki1.1.5 (Figure 7.5).

Figure 7.5 Isometric View of the Three-Dimensional Geological Model for Kamoa

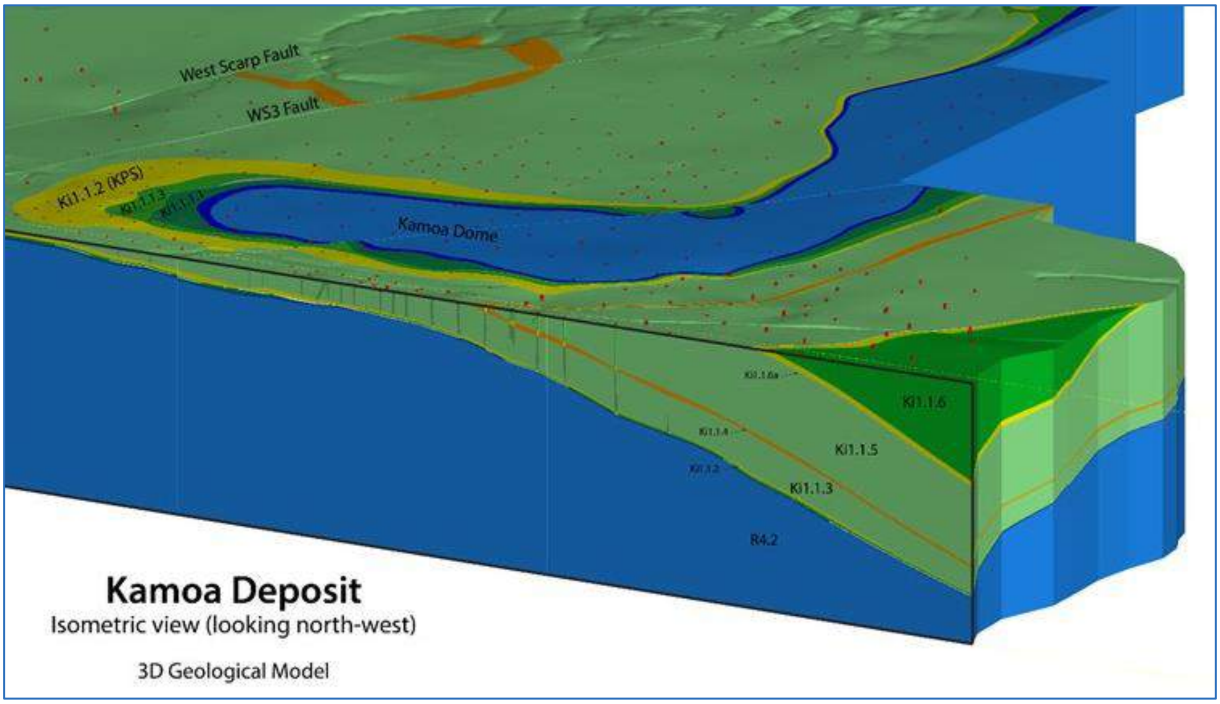
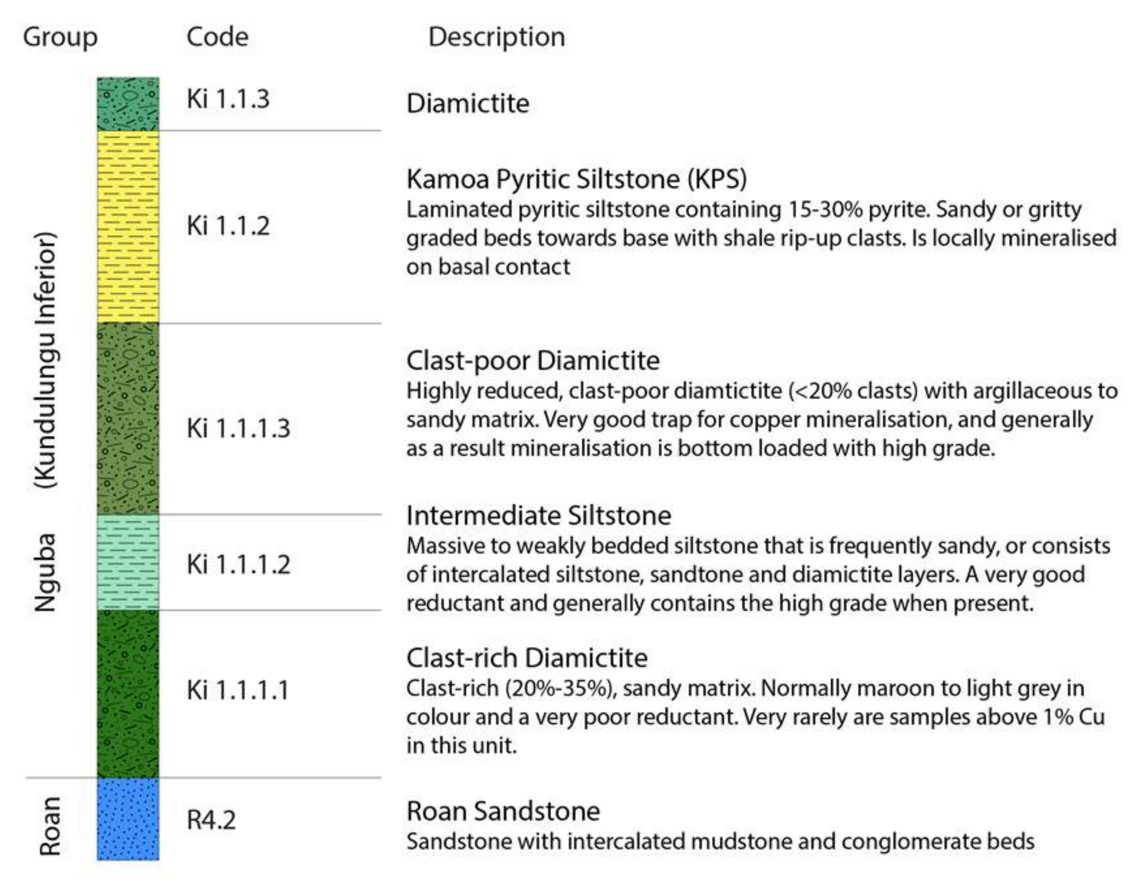


Figure provided by Ivanhoe, 2017. No scale shown given the isometric nature of the image.

As with Kamoa, sandstones of the Mwashya Subgroup of the Roan Group (R4.2) form the basal unit at Kakula. The distinction of clast-rich and clast-poor diamictites at Kakula is not as clear. Kakula is located in an area where the basin has deepened, and the Ki1.1.1 package has thickened dramatically. The diamictites of the Ki1.1.1 are generally clast poor and are typically siltier, suggesting that Kakula represents a more distal depositional environment relative to Kamoa. At Kakula, numerous siltstones are developed within the Ki1.1.1, especially in the lower half of the unit. These siltstones appear to be broadly continuous (Figure 7.9); however, there is no clear correlation between any specific siltstone at Kakula and the intermediate siltstone (Ki1.1.1.2) recognised at Kamoa. A key lithological unit recognised at Kakula is a laterally-continuous basal siltstone, developed just above the R4.2 contact. The basal siltstone is separated from the R4.2 contact by a narrow (often <1 m thick), yet persistently developed, clast-rich diamictite.

Figure 7.6 Local Stratigraphy for the Kamoa Deposit



Diamictite

Kamoa Pyritic Siltstone (KPS)
Laminated pyritic siltstone containing 15-30% pyrite. Sandy or gritty graded beds towards base with shale rip-up clasts. Is locally mineralised on basal contact

Clast-poor Diamictite
Highly reduced, clast-poor diamtictite (<20% clasts) with argillaceous to sandy matrix. Very good trap for copper mineralisation, and generally as a result mineralisation is bottom loaded with high grade.

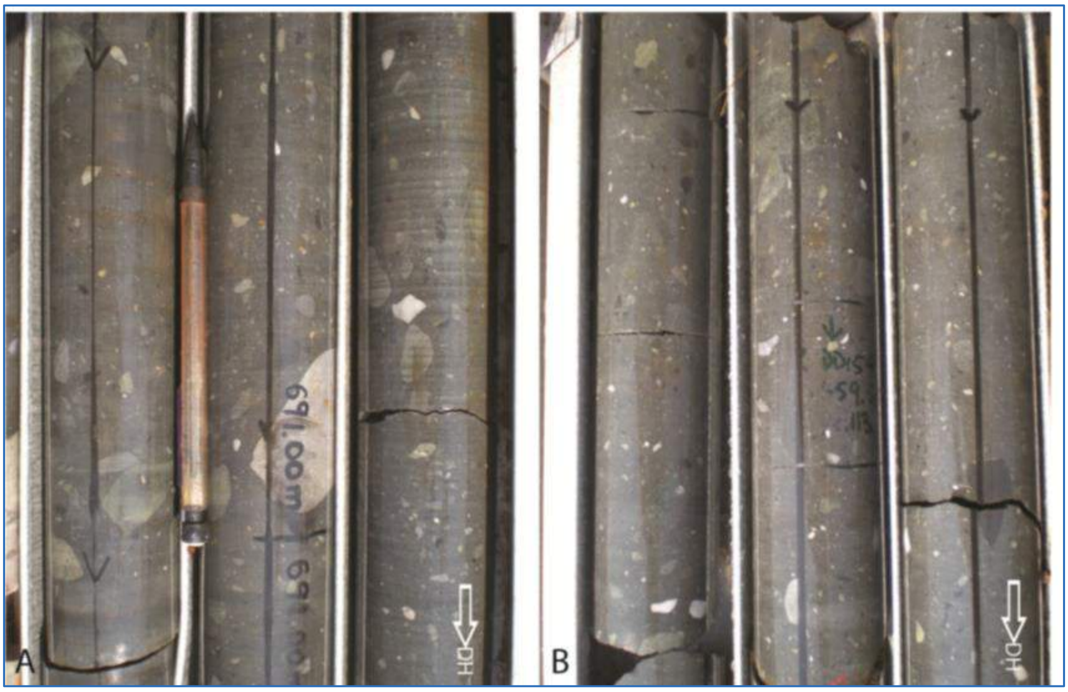
Intermediate Siltstone
Massive to weakly bedded siltstone that is frequently sandy, or consists of intercalated siltstone, sandtone and diamictite layers. A very good reductant and generally contains the high grade when present.

Clast-rich Diamictite
Clast-rich (20%-35%), sandy matrix. Normally maroon to light grey in colour and a very poor reductant. Very rarely are samples above 1% Cu in this unit.

Roan Sandstone
Sandstone with intercalated mudstone and conglomerate beds

Figure provided by Ivanhoe, 2016.

Figure 7.7 Kamoa Clast-rich Diamictite (A) and Clast-poor Diamictite (B) DKMC_DD159



Source: Schmandt et al (2013).

Figure 7.8 Distinctive Varves, Dropstone, and Pyrite at the Base of the KPS (DKMC_DD154) at Kamoa

Source: Schmandt et al (2013).

Figure 7.9 North-West to South-East Section Through Kakula Illustrating the Numerous Siltstone Units Developed Towards the Base of the Ki1.1.1.

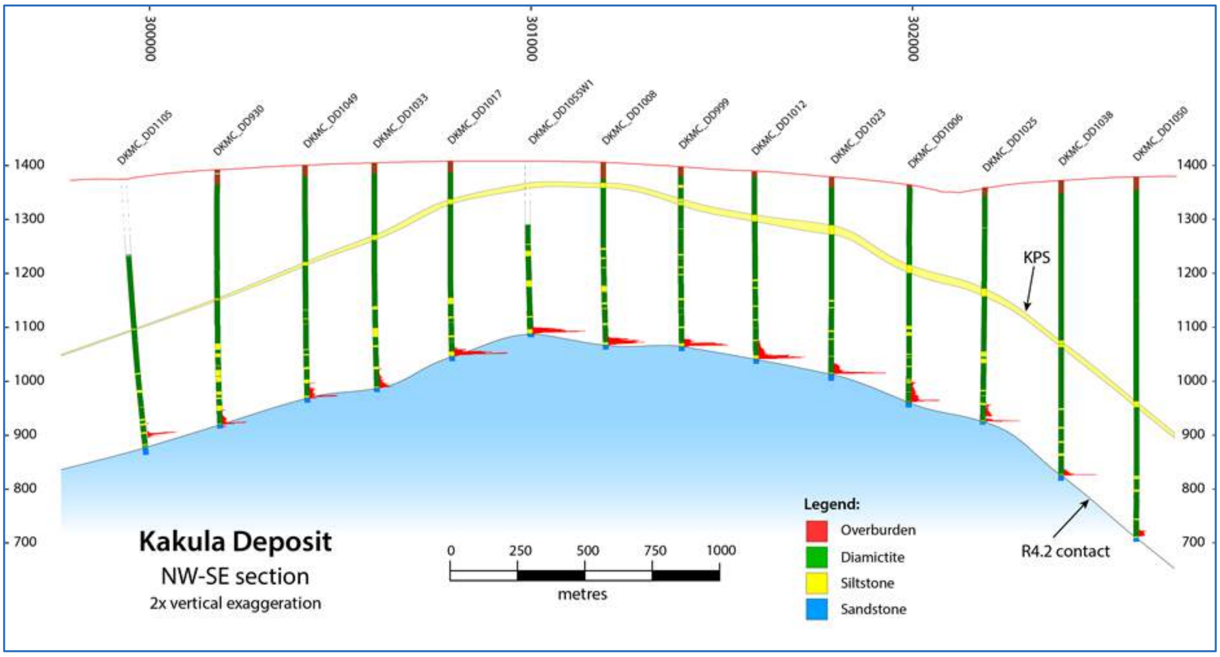


Figure provided by Ivanhoe, 2017. Red bars indicate assay intervals grading ≥ 0.5% Cu.

7.3.2 Thicknesses of Model Units

Vertical thickness plots for the two diamictite units and the KPS, (Figure 7.10 to Figure 7.12), indicate a north-west trend to stratigraphic thickness developed across the Project area. This is particularly evident in Kansoko Sud, where changes in thickness of the Ki1.1.1.1 and Ki1.1.1.3 are evident across what has been identified as the Mupaka Fault; a syn-sedimentary growth fault (Twite et al., 2019). The thickening is very obvious on a section line perpendicular to the thickening orientation, refer to Figure 7.13.

The intermediate siltstone (Ki1.1.1.2) shown in Figure 7.14 and Figure 7.15 is either absent or locally developed as a single siltstone unit separating the clast-poor and clast rich diamicties. When developed, the Ki1.1.1.2 siltstone is typically preferentially mineralised.

In the south-west, the thickening of the diamictite units is also marked by the development of thicker siltstone sandstone siltstone units, or the development of numerous siltstone units, comparable to the numerous siltstone units identified within the Ki1.1.1 at Kakula.

Figure 7.10 Ki1.1.1.1 Vertical Thickness

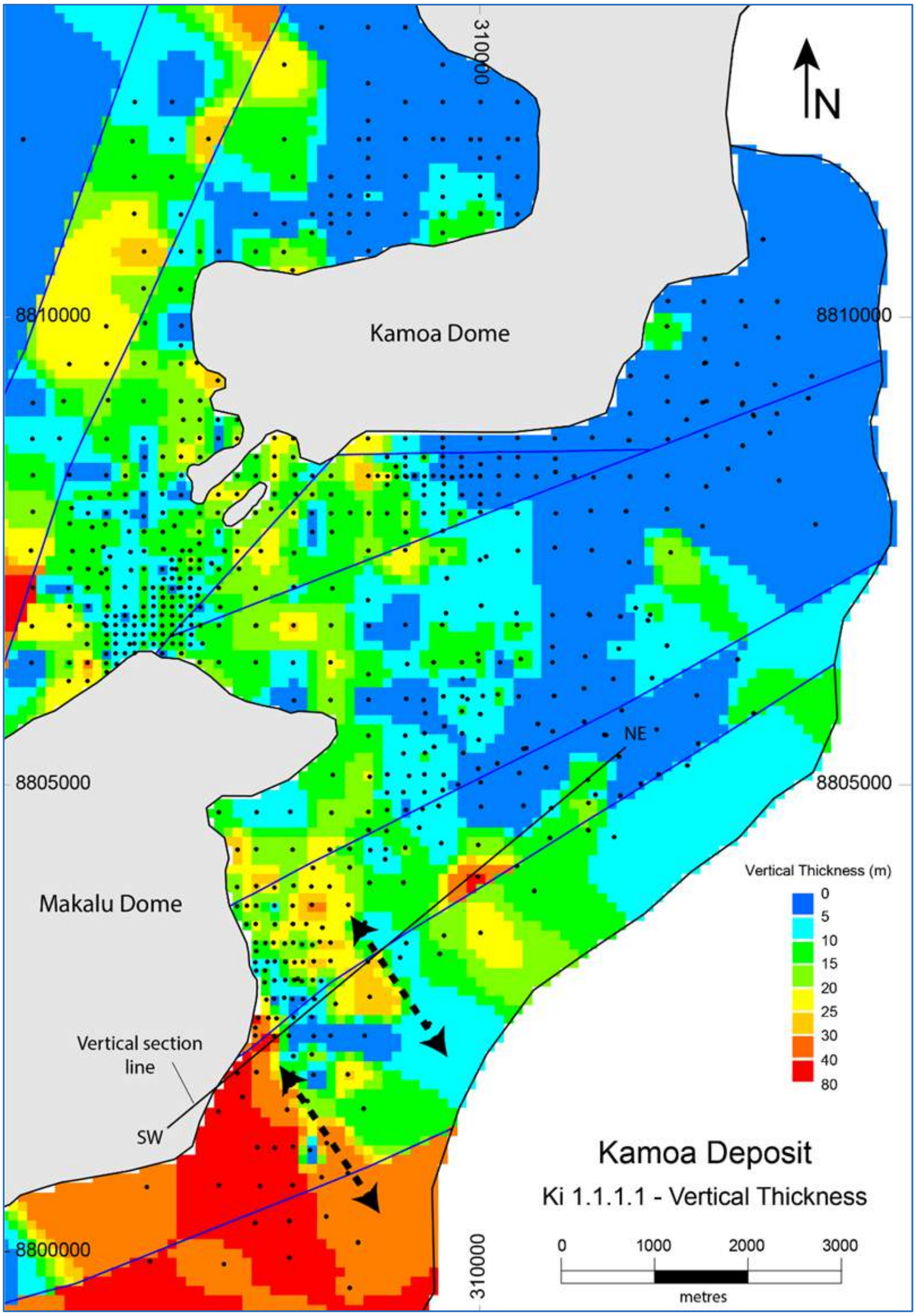


Figure provided by Ivanhoe, 2016; black line is the trace of the cross-section shown in Figure 7.13; grey areas represent domes or leached zones; blue lines are interpreted faults; black dashed lines represent north-west oriented thickness trend.

Figure 7.11 Ki 1.1.1.3 Vertical Thickness

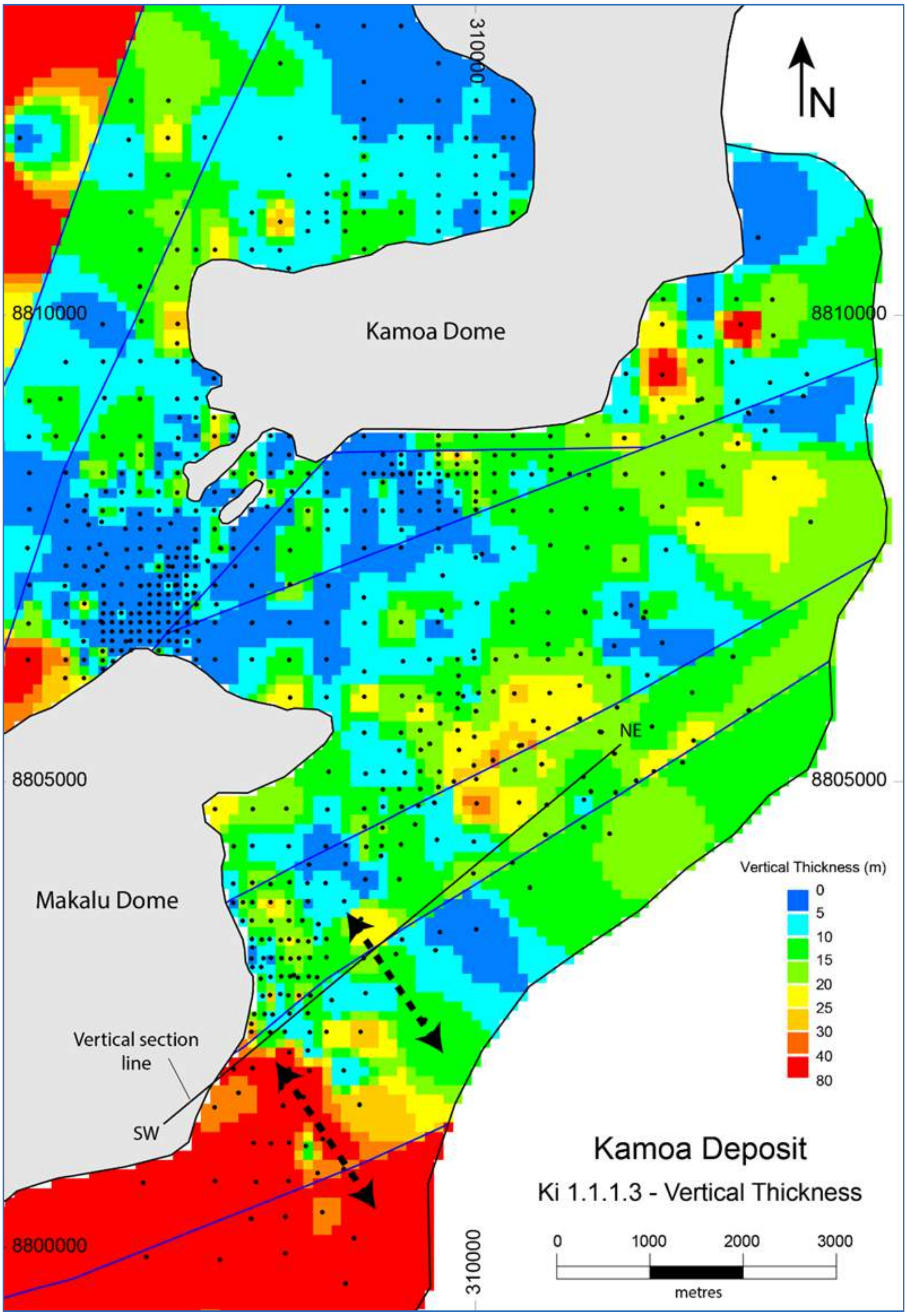


Figure provided by Ivanhoe, 2016; black line is the trace of the cross-section shown in Figure 7.13; grey areas represent domes or leached zones; blue lines are interpreted faults; dashed black lines represent north-west oriented thickness trend.

Figure 7.12 Ki 1.1.2 Vertical Thickness (KPS)

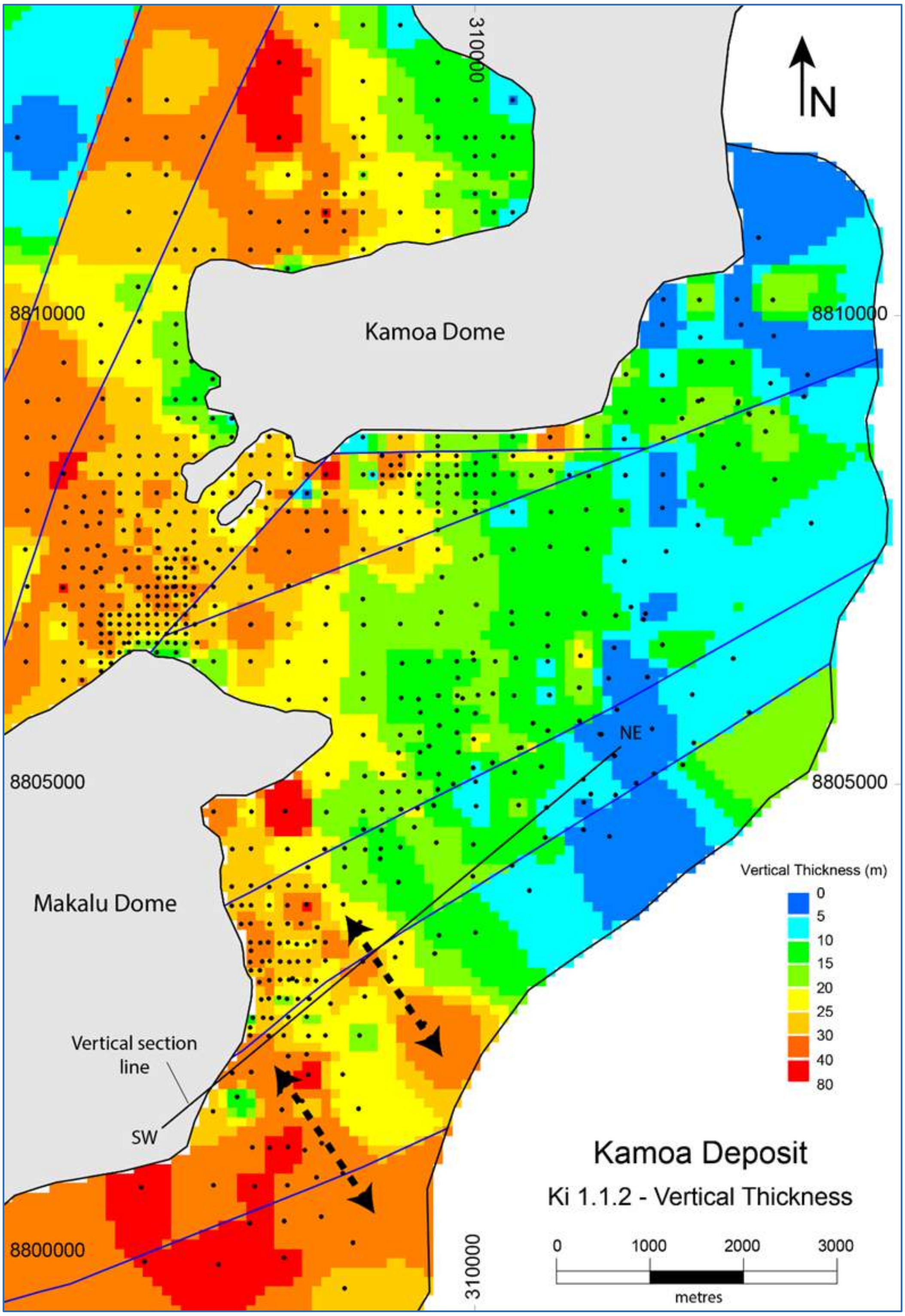


Figure provided by Ivanhoe, 2016; black line is the trace of the cross-section shown in Figure 7.13; grey areas represent domes or leached zones; blue lines are interpreted faults; dashed black lines represent north-west oriented thickness trend.

Figure 7.13 Section from Kansoko Sud (SW) to Kansoko Centrale (NW)

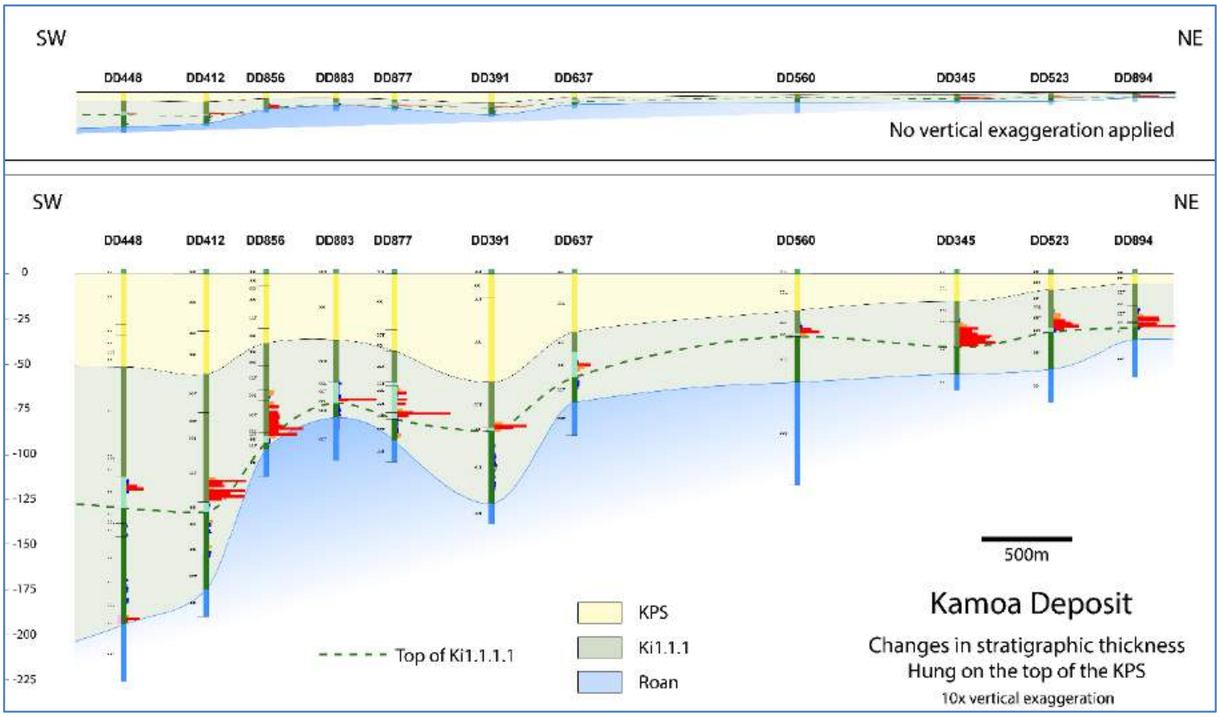


Figure provided by Ivanhoe, 2014, illustrating the thickening of units to the south-west; section line location is indicated in Figure 7.10 to Figure 7.12. Copper grades are shown as histograms with red being >1% TCu.

Figure 7.14 Ki 1.1.1.2 Vertical Thickness

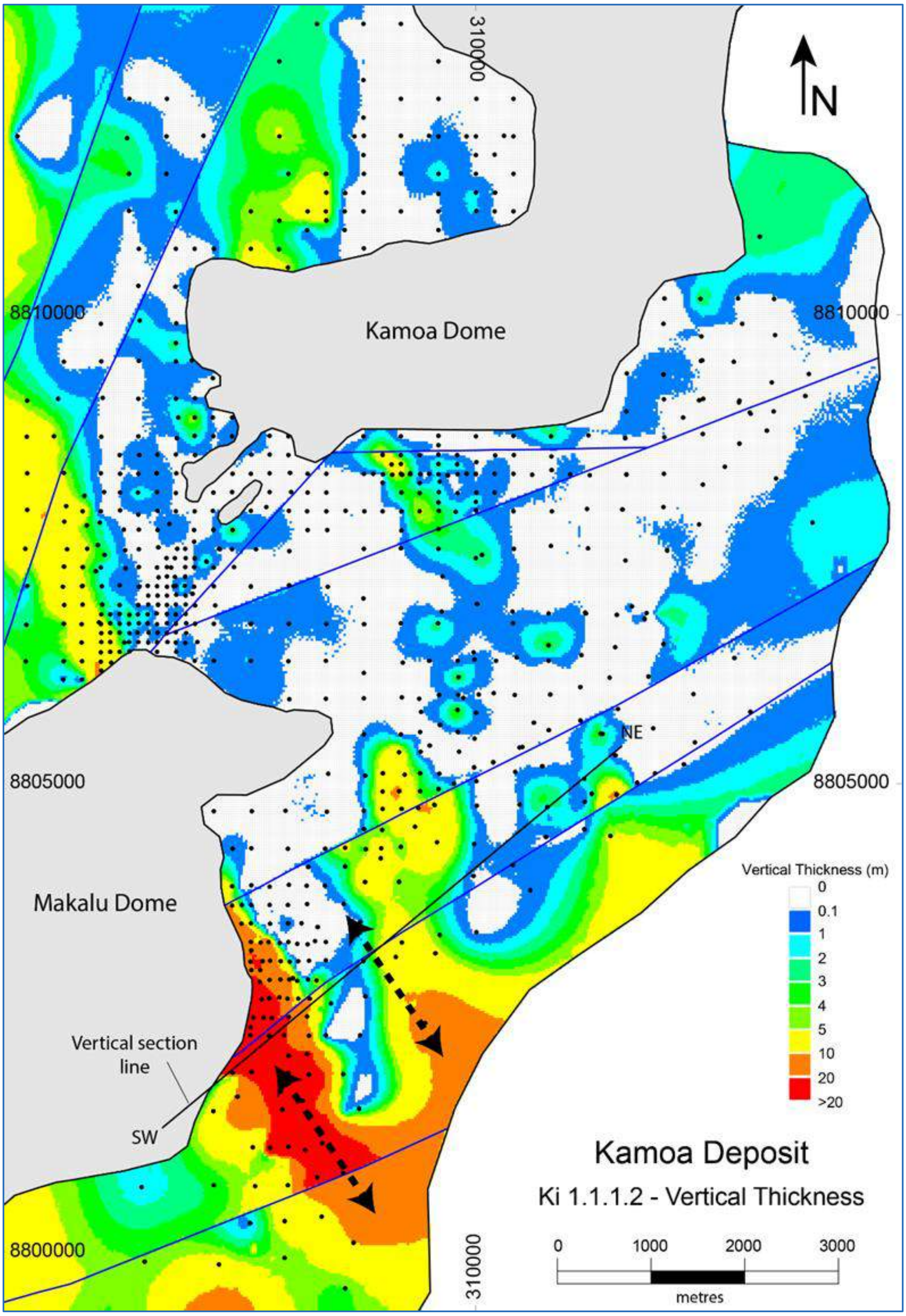


Figure provided by Ivanhoe, 2016; black line is cross-section shown in Figure 7.13; grey areas represent domes or leached zones; blue lines are interpreted faults; dashed black lines represent north-west oriented thickness trend.

Figure 7.15 Occurrence of Ki 1.1.1.2 Intermediate Siltstone Units within the Ki 1.1.1

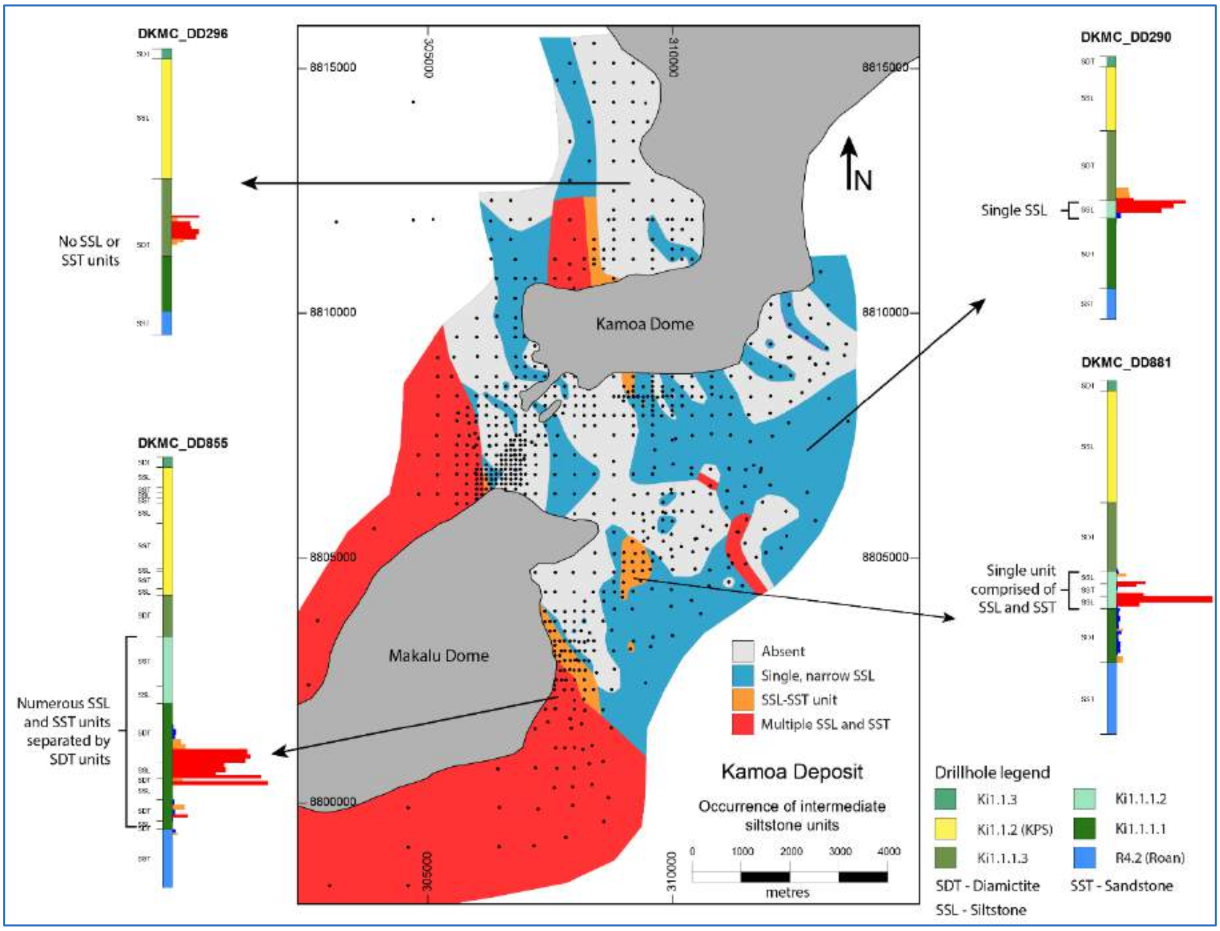


Figure provided by Ivanhoe, 2018. Copper grades, shown as red histograms (where >1% TCu).

Figure 7.16 Modelled Ki 1.1.1.2 Intermediate Siltstone Sub-Units South-East of the Makalu Dome

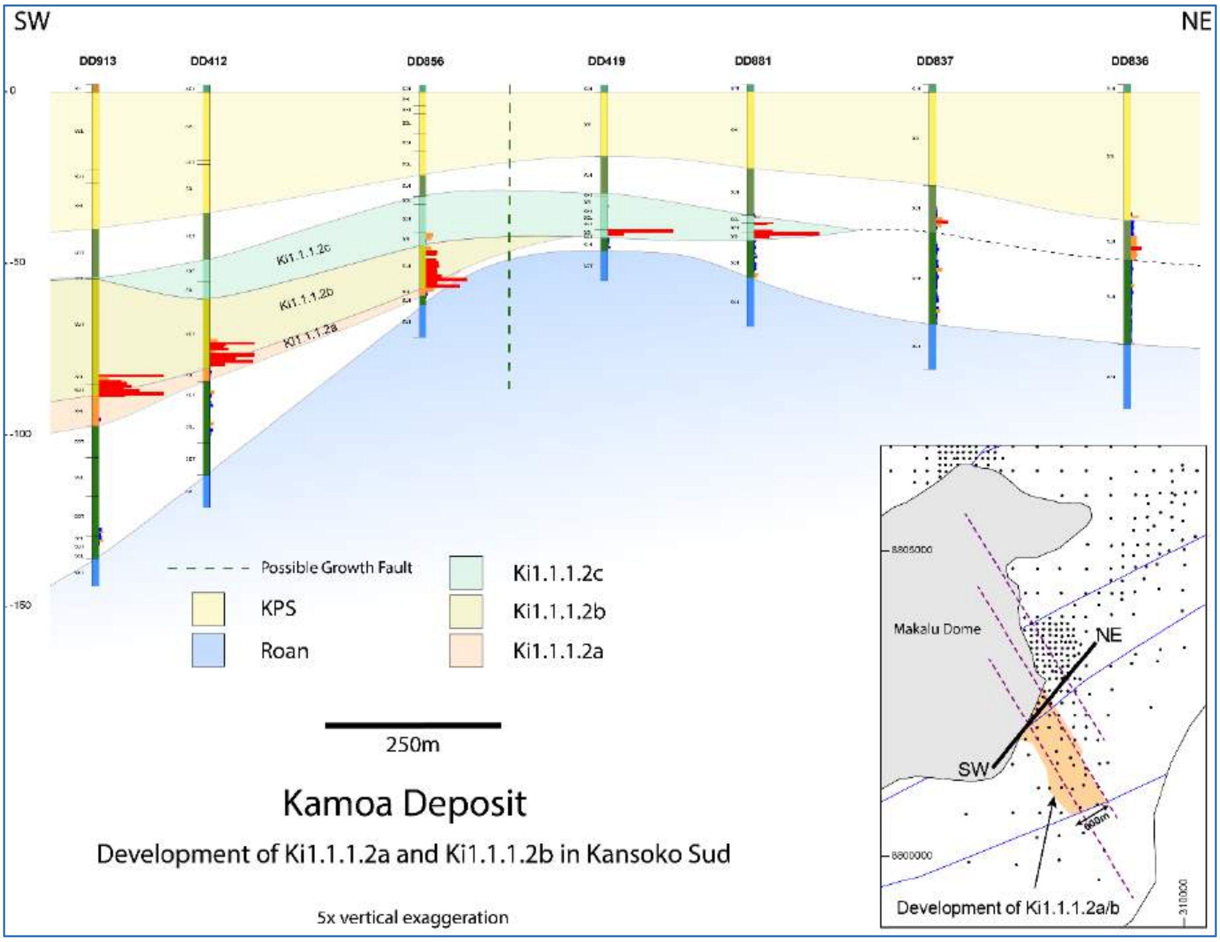


Figure provided by Ivanhoe, 2017. Copper grades in percent, shown as red histograms if >1% TCu.

In the central portions of Kakula, a strong correlation is evident between the presence of the basal siltstone developed within the Ki1.1.1 and the development of high-grade mineralisation. The vertical thickness of the basal siltstone is thickest in the shallowest parts of the deposit, with a very strong alignment along a trend striking approximately 120° (Figure 7.17).

The KPS at Kakula crops out in the vicinity of the domes, preventing the determination of a thickness for the Ki1.1.1 where the KPS has been entirely eroded. The Ki1.1.1 generally thickens to the west. The Ki1.1.1 is considerably thicker than at Kamoa, with vertical thicknesses varying from 180 m to over 400 m at Kakula West (Figure 7.18). A pronounced north-east orientation in thickness trends is evident at Kakula West, and this observation has been incorporated into the search orientations used during grade estimation.

Figure 7.17 Vertical Thickness of the Basal Siltstone within the Ki1.1.1 at the Kakula Deposit

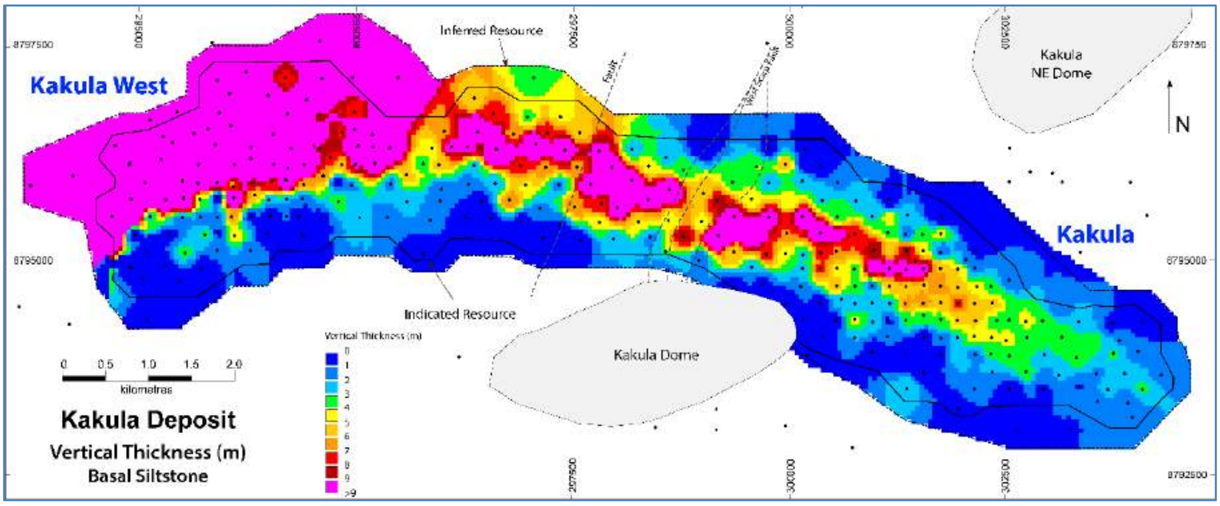


Figure provided by Ivanhoe, 2019. Vertical thickness estimated using an isotropic search.

Figure 7.18 Vertical Thickness of the Ki1.1.1 at the Kakula Deposit

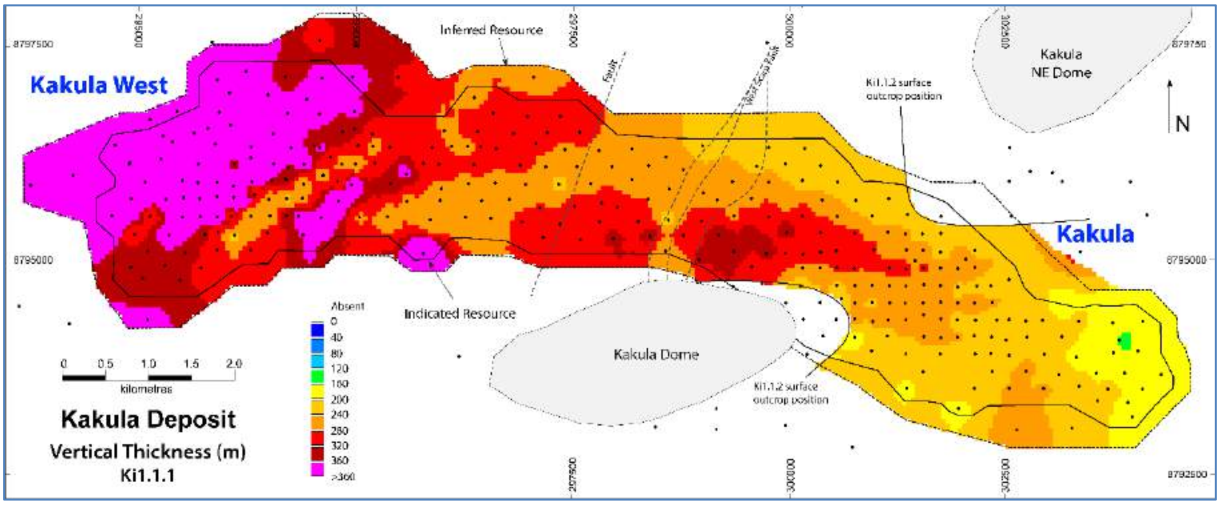


Figure provided by Ivanhoe, 2019. Vertical thickness estimated using an isotropic search.

In contrast to the general thickening of stratigraphic units to the south-west observed at Kamoa, the KPS at Kakula is much thinner. Thickening of the KPS in the central portions of Kakula occurs along the same 120° trend (Figure 7.19), although it is offset relative to the thickening observed in the basal siltstone. Complicated sedimentation patterns are evident at Kakula West due to proximity to the north-east trending extensional faults, and east-north-east transfer faults active during sedimentation.

There appears to be no obvious control on thicknesses of stratigraphic or lithological units relative to modelled brittle faults. These faults, part of the West Scarp Fault system, appear to be later structures that offset the different units.

Figure 7.19 Vertical Thickness of the Ki1.1.2 (KPS) at the Kakula Deposit

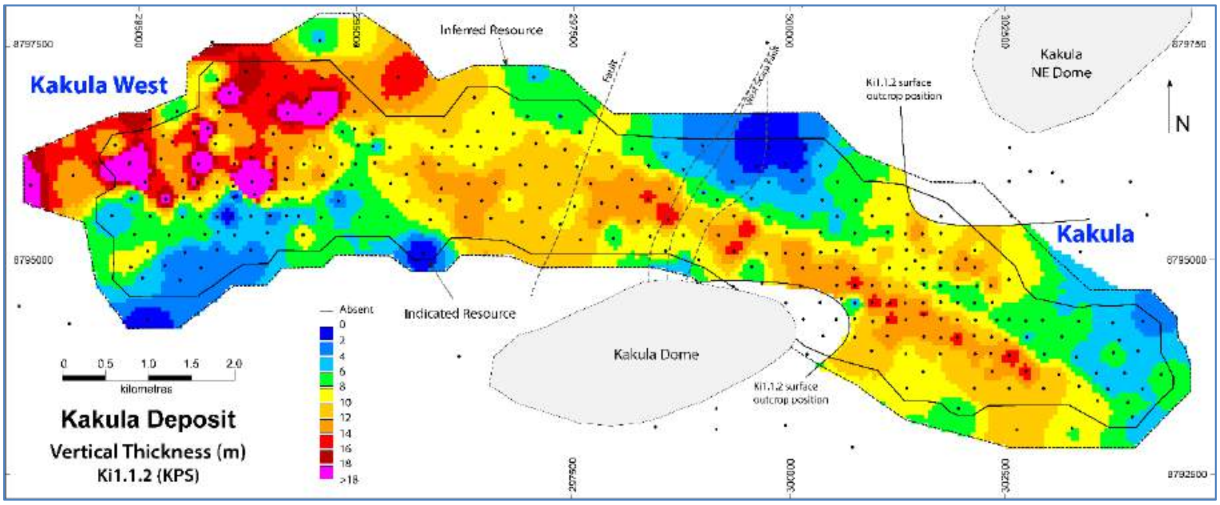


Figure provided by Ivanhoe, 2019. Vertical thickness estimated using an isotropic search.

7.3.3 Igneous Rocks

Andesite/dolerite sills have been identified in the Project area. These occur as one or more, 5 m to 80 m thick, apparently concordant tabular bodies in the extreme north-east of the Project area.

7.3.4 Structure

Geophysical data (primarily the magnetics second vertical derivative) and topographic expression provide the primary support for continuity of structural features, whilst the drillhole data and geotechnical logging provide supporting evidence (refer to Figure 7.20).

Ongoing interpretation has sought to develop a broader structural framework for the Kamoa deposit. The 2014 structural model for Kamoa consists of 31 faults divided into six sets of differing orientations. A simplified subset of significant faults (those expecting to exhibit offset >10 m) have been incorporated into the January 2018 resource model. These structures were used as boundaries to divide the mineralisation into structural zones, refer to Figure 7.21.

Figure 7.20 Structural Model Overlaid on Second Vertical Derivative Magnetic Image

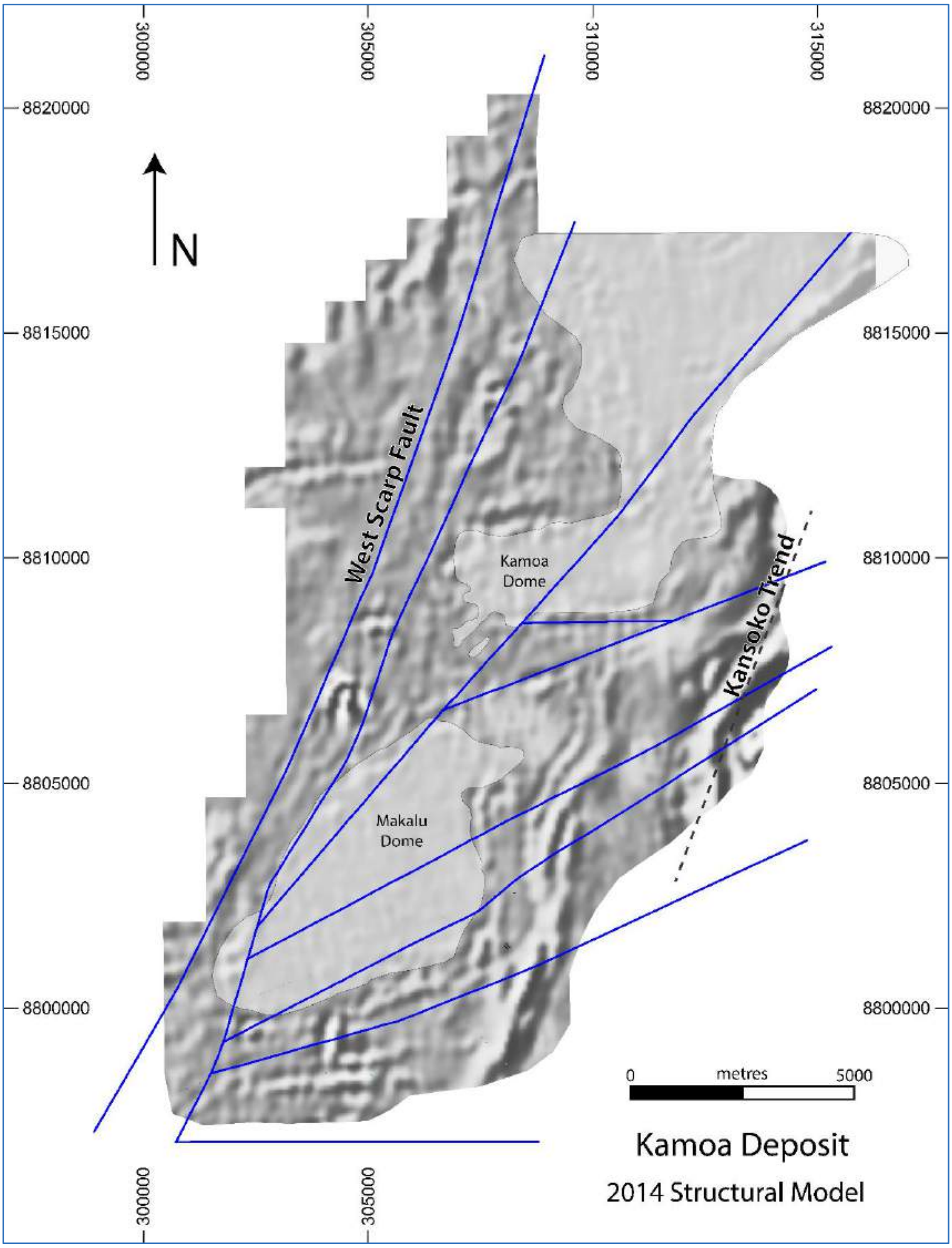


Figure provided by Ivanhoe, 2016; blue lines are interpreted fault traces; light grey areas are domes and surrounding leached zones.

Figure 7.21 Structural Model and Contours (masl) for the 1.5% TCu Mineralised Zone at the Kamoa Deposit

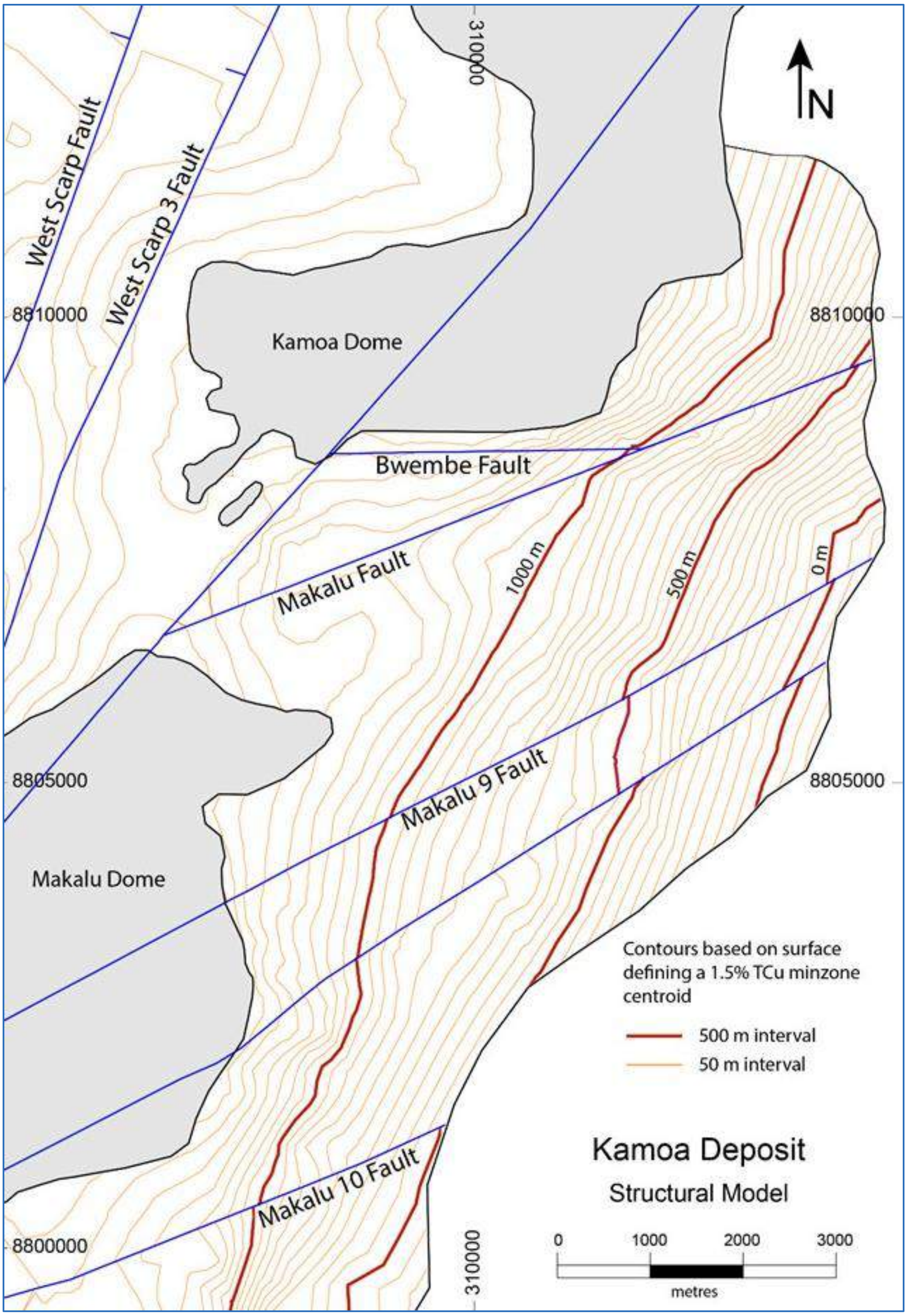


Figure provided by Ivanhoe, 2018.

The presence of very open folds at the Kamoa deposit are believed to account for offsets observed between drillholes that are not attributed to faults. Two sets of fold axes are observed, with one set striking approximately north–south and the second set striking west–east, or north-east. The intersection of these two orientations accounts for the domes and their undulations in shape.

The fault sets are interpreted to relate to one of three deformational events:

- D1: Crustal transtension forming the Kundelungu Rift (735–645 Ma) (De Waele et al 2005). Nguba sediments accumulated into an extensional basin, with sedimentation controlled by active faults.
- D2: Compression during the development of foreland basin systems (550–520 Ma) (Johnson et al 2005) led to the development of gentle folding throughout the area and creation of domes.
- D3: East–west extension, forming cross-cutting, north-striking normal faults, which truncate the western edge of the area. The West Scarp Fault is the most prominent of these features. The West Scarp Fault has a west-side down-throw of approximately 200 m to 400 m. The effect of this fault is clearly evident in the topographic image (refer to Figure 7.22).

Microstructures are commonly observed in core, particularly in the finely laminated siltstone units. Observed offsets are on the millimetre or centimetre scale, with either normal or reverse sense of movements identified. A steep to vertical foliation is defined by the alignment of clasts or minerals within the matrix, and the alignment of fine and coarse-grained sulphides. In rare cases, unusually steep bedding is identified to occur over intervals of 0.5 m to 2 m. These occurrences often coincide with the high copper grades (>5% TCu) and have been observed to align on the north–north-west growth fault trend evident from changes in thickness of individual stratigraphic units. In these areas, microstructures are usually flat-lying, suggesting they formed earlier and were rotated during the time the bedding was steepened. The foliation, however, remains steep regardless of the nature of the bedding, suggesting it is a later overprint (Twite, 2019). Examples of these features are shown in Figure 7.23.

Syn-sedimentary normal faults have been documented as first-order ore controls at several de-posits in the Zambian Copperbelt, such as Mwambashi B, Chambishi, Konkola, Musoshi and Fishtie (Selley et al. 2005, Hendrickson et al. 2015).

Figure 7.22 Structural Influences on Topography

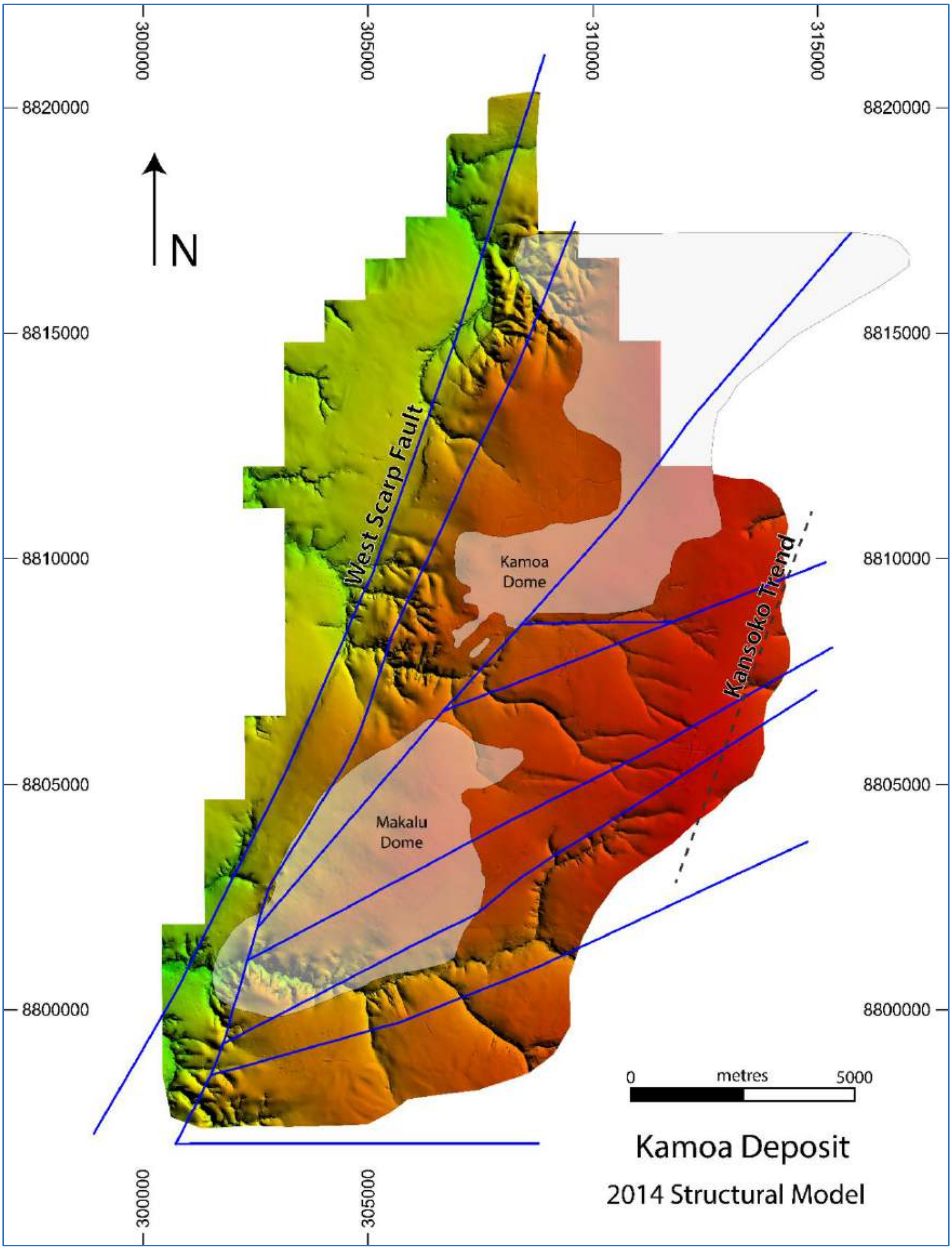


Figure provided by Ivanhoe, 2016.

Figure 7.23 Microstructural Features Evident at Kamoa. Normal and Reverse Offsets (left), and Steep Bedding and Foliation (S_1) (right)

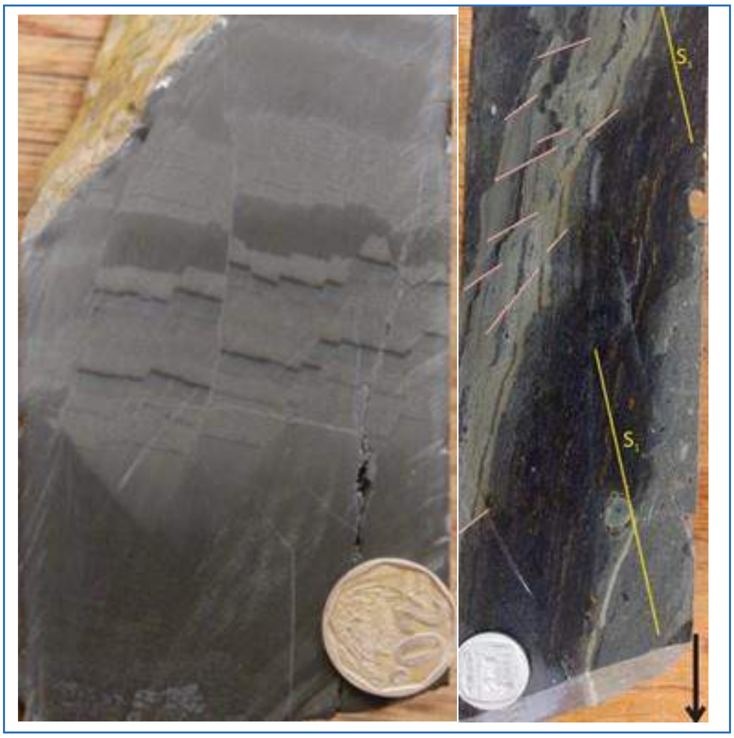


Figure provided by Ivanhoe, 2016. Yellow line indicates foliation orientation. Whitish-pink lines identify microstructures offsetting layering.

Kakula's current geometry is strongly influenced by extensional faults active at the time of deposition, which formed a number of sub-basins across the axis of a broad doubly plunging antiform (dome). Depositional conditions at the time of sub-basin formation led to the development of a laterally continuous siltstone layer.

Lithological units are observed to drape across these extensional faults, rather than have discrete offsets. Three broad zones are evident from the current drilling, and have aided the choice of search orientation during resource estimation:

- In the south-eastern and central portions of the deposit, limited development of the extensional faults is observed.
- At Kakula West, a series of sub-basins have been formed adjacent to extensional faults striking north-east and east-north-east. Draping of stratigraphic units over these extensional faults at the Ki1.1.1–R4.2 boundary can occur across elevation differences exceeding 50 m.
- On the western edge of Kakula West, pronounced extensional faults trending north-east are evident, with elevation differences exceeding 200 m (west block down) in some areas.

Strain associated with basin inversion during the Lufilian appears relatively low. In general the relative structural positions and elevations of the original sub-basins and intervening regional 'highs' have been preserved. Inversion appears to have had the principal effect of producing low amplitude folds, while amplifying and tightening the 'drapes' across the inverted normal faults. A strong foliation parallels the elongated dome structure at Kakula West, particularly where the Ki1.1.2 is close to surface.

Younger brittle structures are also observed at Kakula that offset the mineralisation. The most prominent faults are north–north-east trending structures including, and related to, the West Scarp Fault. Four discrete structures have been modelled (Figure 7.24). The West Scarp Fault and a second fault approximately 150 m to the west, account for the majority of the offset observed, and are evident in drill core and from surface modelling (most notably in DKMC_DD1080). These two faults are steeply dipping (approximately 75° to the west) normal faults (west block down) and jointly account for offset of 150 m to 200 m. Approximately 400 m east of the West Scarp Fault, a reverse fault has been modelled dipping approximately 75° to the west, with a reverse offset (east block down) of approximately 10 m. Approximately 1,500 m west of the West Scarp Fault a steeply dipping (approximately 75° to the east) normal fault (east block down) has been modelled based on features observed in the core and evidence in the alignment of topographic features.

These structures are known from Kamoa and are evident in discontinuities in magnetic signatures. They are considered related to one another given their orientation, proximity to one another, age relationship to other features (they are all young) and by their characteristics in the core, where steep breccias, calcite veining and broken core are evident. No true thicknesses of these fault zones have yet been attained in drilling. Additional drilling and modelling are planned to further characterise these faults.

Additional observed structures in drill core include steeply-dipping chaotic breccias and gouges (Figure 7.25). Cohesive "crackle" breccias (a breccia having fragments parted by planes of rupture but showing little or no displacement (Norton, 1917)) are also developed. A flat-lying cohesive breccia occur close to the Ki1.1.1–R4.2 contact. It ranges in thickness from 15 cm to 90 cm and is frequently well mineralised (Figure 7.26).

Figure 7.24 Long Section of the North-West Kakula Area Illustrating Offset Across the Modelled Faults

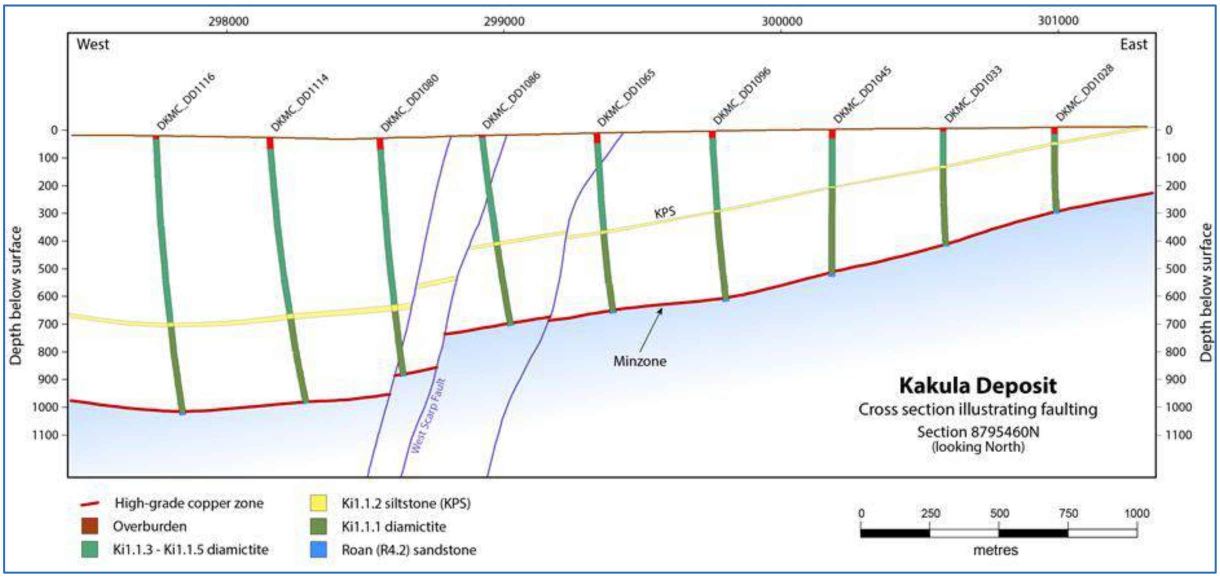


Figure provided by Ivanhoe, 2018.

Figure 7.25 Steeply-Dipping Chaotic Breccia in DKMC_DD1015 (at 244.4 m)

Figure provided by Ivanhoe, 2016.

Figure 7.26 Massive Chalcocite Band Towards the Base of the Mineralised Zone in DKMC_DD1009 (at 354.1 m)

Figure provided by Ivanhoe, 2016.

The structural model used in construction of the Kakula Mineral Resource estimate includes the four north–north-east structures. The four modelled faults have resulted in the Mineral Resource area being divided into five structural blocks, where offsets in the elevation of the stratigraphic and mineralised zones are modelled. Figure 7.27 shows the contours and offsets for the Roan–-Ki1.1.1 contact. Drilling has confirmed that the edges of the Kakula and Kakula West domes are characterised by areas of pronounced steeper dip resulting either from syn-deposition faulting or uplift during inversion.

At Kamoa, the stratigraphic units generally dip gently away from the dome edges at between 5° to 20°. The Kamoa and Kamoa Nord areas are particularly gently-dipping; Kansoko Sud and Kansoko Centrale generally dip at between 10° to 20° to the south-east, with occasional steepening up to 30°. The steepest-dipping portions of the deposit are in Kansoko Nord, where units dip to the south or south-east at 15° to 40°.

The shallowest portion of the Kakula deposit between the two domes (Figure 7.27) is also gently-dipping (<10°). To the west, dips gradually increase up to 15° towards the West Scarp Fault. To the east, the dip increases to >35° at the eastern edge of the resource estimate area.

Figure 7.27 Structure Model for the Kakula Resource Area Showing Contours (masl) for theKi1.1.1–R4.2 (Roan) contact

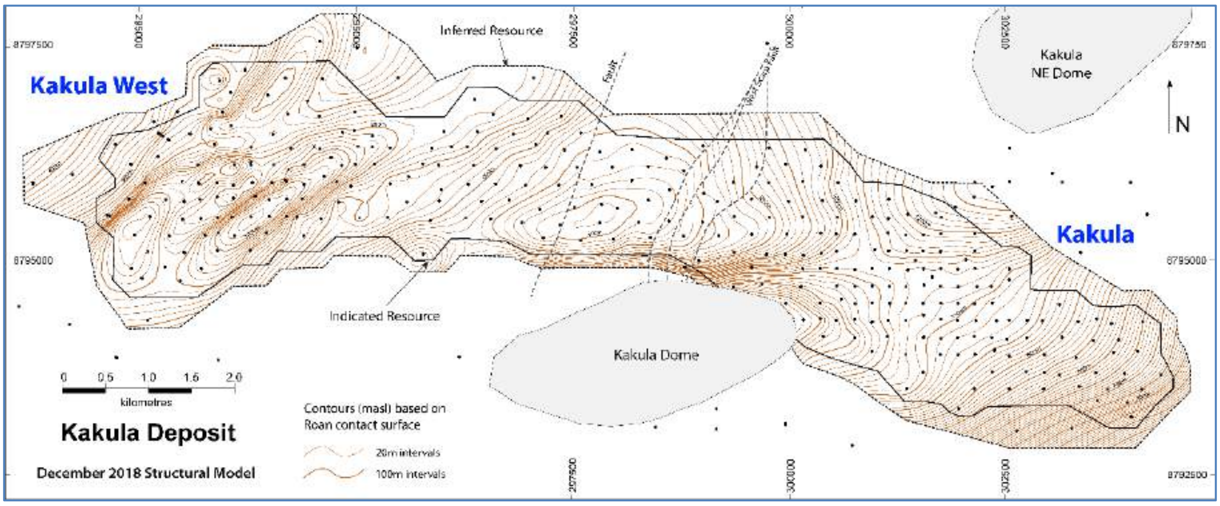


Figure provided by Ivanhoe, 2019.

7.3.5 Metamorphism

The Katangan rocks in the Project area contain chlorite, and are weakly metamorphosed to lower greenschist facies.

7.3.6 Alteration

Alteration in sediment-hosted copper deposits is typically subtle, and comprises low-temperature diagenetic minerals. At Kamoa, core logging indicates that alteration minerals include carbonate, chlorite, sericite, potassium feldspar, and haematite. Carbonate occurs in minor amounts in the Ki1.1.1 rocks, as up to 5% approximately 1 mm size disseminated rhombohedra.

The matrix to the Ki1.1.1.1 sandy clast-poor diamictite weathers to a pale beige/buff colour, suggestive of fine-grained, slightly ferroan dolomite in the matrix. The footwall R4.2 sandstones contain disseminated, and patchy to lensoidal, dolomite–calcite, commonly pinkish in colouration. A later, overprinting, bleached, probably albitic dolomitic, alteration is locally present adjacent to quartz–carbonate–sulphide veins near the West Scarp Fault.

7.3.7 Mineralisation

Mineralisation at the Kamoa-Kakula Project has been defined over an irregularly shaped area of 27 km x 21 km. Mineralisation is typically stratiform, and vertically zoned from the base upward with chalcocite (Cu_2S), bornite (Cu_5FeS_4) and chalcopyrite ($CuFeS_2$). There is significant pyrite mineralisation above the mineralised horizon that could possibly be exploited to produce pyrite concentrates for sulphuric acid production (needed at oxide copper mines in the DRC).

The dip of the mineralised body ranges from 0° to 10°, to 15° to 20° on the flanks of the domes. At Kamoa mineralisation thicknesses at a 1.0% Cu cut-off grade ranges from 2.3 m to 15.8 m (for Indicated Mineral Resources). The deposit has been tested locally from below surface to depths of more than 1,560 m, and remains open to the west, east, and south. At Kakula, mineralisation thicknesses at a 1.0% Cu cut–off grade range from 2.9 m to 42.5 m (for indicated Mineral Resources). The deposit has been tested locally from below surface to depths of more than 1,000 m, and remains open to the south-east and west.

Mineralisation in the majority of the Katangan Copperbelt orebodies such as at those located at Kolwezi and Tenke–Fungurume is oxide in nature and is hosted in the Mines subgroup (R2).

The mineralisation at Kamoa-Kakula differs from these deposits in that it is primarily sulphide mineralisation located in the Grand Conglomerat unit (Ki1.1) at the base of the Lower Kundelungu Group. In contrast to the neighbouring Kolwezi deposits, mineralisation at Kamoa-Kakula is characterised by a lack of cobalt (Schmandt et al, 2013). Very little oxide mineralisation is evident at Kamoa or Kakula, likely due to the leaching effects of weathering of the thick pyritic KPS overlying the mineralised zone. Close to dome edges, where the mineralisation nears surface, total or partial leaching of the copper sulphides has occurred. Relatively laterally narrow zones of supergene enrichment are also observed in these areas; however, the bulk of the copper mineralisation is hypogene. The change from supergene to hypogene is generally transitional, with a strongly developed vertical zonation evident in the hypogene. Locally there is oxide copper mineralisation (malachite) developed at depth within the hypogene zone along faults and fractures.

The genetic model developed by Ivanhoe reflects modern interpretations for formation of the Copperbelt. During basin closure and broad folding, oxidizing saline brines migrated up dip through porous Roan sandstone and leached copper. The brines encountered a redox boundary at the base of a diamictite, the Grand Conglomerat. Regionally, the diamictite of the Lower Kundulungu formed a redox boundary, causing the precipitation of copper sulphide minerals. At Kamoa, the clast-rich diamictite (Ki1.1.1.1) is considered to be only weakly reducing, and thus generally hosts only low-grade (<0.5% TCu) mineralisation. The intermediate siltstone (Ki1.1.1.2) and clast-poor diamictite (Ki1.1.1.3) are considered to represent significantly better reducing horizons and thus host the majority of the primary mineralised zone. Some of the most consistent and highest-grade intervals are intersected where the clast-rich diamictite is absent, and the clast-poor diamictite rests directly on the Roan contact.

At Kakula, the narrow (<3 m) clast-rich diamictite immediately above the Roan contact is only weakly reducing and thus has low copper grades. The basal siltstone overlying the clast-rich diamictite is a very strong reductant and accounts for the majority of very high-grades (>6% Cu). The lateral continuity of this reductant allows for the unique continuity of grades >6% TCu at Kakula. The diamictite overlying the basal siltstone is clast-poor and is also a good reductant; however it hosts low-grade copper mineralisation relative to the basal siltstone.

The earliest sulphide mineralisation at Kamoa-Kakula was deposited during diagenesis and formed abundant framboidal and cubic pyrite in the laminated siltstones (particularly the KPS) (Schmandt et al, 2013).

Mineral Zonation

Two broad categories of lateral zonation are evident at Kamoa (hypogene and supergene); however, within the hypogene, additional lateral zonation is evident based on the relative abundance of chalcopyrite, bornite and chalcocite. The dominant sulphide species within the mineralised zone is interpreted to be a lateral mineral zonation. The change from supergene to hypogene is generally transitional with a strongly developed vertical zonation evident in the hypogene (refer to Figure 7.28).

Kakula shows similar mineral zonation, but the mineralisation is mainly hypogene chalcocite dominant. Bornite and chalcopyrite zones are not as well developed as at Kamoa, and supergene chalcocite zones do not occur at Kakula.

Figure 7.28 Schematic of Mineral Zonation at Kamoa

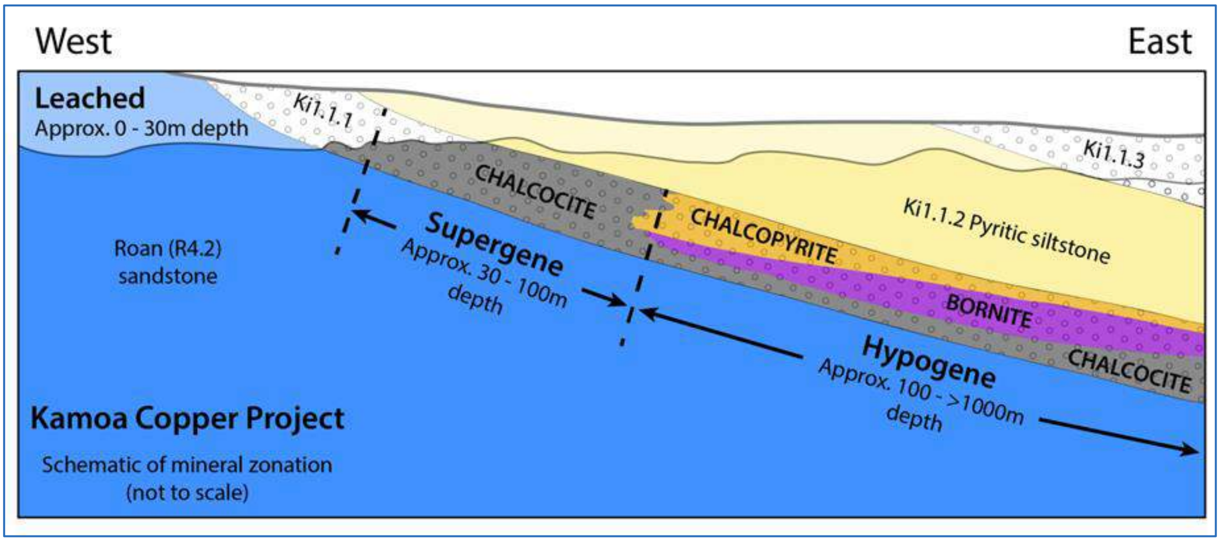


Figure is schematic and not to scale. Leached zone ranges from 0 to 30 m vertical depth from surface. Supergene zone ranges from 30 m to 100 m vertical depth from surface. Hypogene typically extends from 100 m vertical depth to more than 1,560 m. The supergene/hypogene interface is typically at less than 100 m depth; locally it can be deeper in the vicinity of faults and fractures. Figure by Ivanhoe, 2014.

Supergene copper mineralogy is dominated by fine-grained chalcocite with secondary native copper and cuprite. The supergene zone may extend to depths of 250 m or more along fracture zones and stratigraphic contacts (Schmandt et al, 2013).

Mineralising fluids in the Copperbelt have been modelled by Muchez & Corbella (2016) to advance slowly, laterally and across stratigraphic layers away from the fluid source, driving the reactions between a copper-rich fluid and the host rock reductant (anhydrite or pyrite), precipitating copper sulphides. This mineralising front is also interpreted to be capable of dissolving some of the earlier copper sulphides, with the stratigraphically lower, more copper-rich sulphides partially replacing the overlying, more copper-poor sulphides (Twite et al. 2018). This leads to a characteristic zonation whereby pyrite is replaced by chalcopyrite, chalcopyrite by bornite, and bornite by chalcocite. This zonation is evident at Kamoa-Kakula and elsewhere on the Copperbelt (Van Langendonck et al. 2013).

At Kamoa, chalcopyrite dominates, primarily as fine-grained disseminations in the diamictite matrix, although very coarse chalcopyrite can form as elongated grains up to 5 mm in length rimming clasts or fragments, or defining strain shadows to clasts (Figure 7.29). A steep to vertical foliation is defined by the weak alignment of clasts and minerals within the matrix, but is often best displayed by the alignment of fine and coarse-grained sulphides (Twite, 2016; Figure 7.29). Bornite is typically fine-grained and disseminated in the matrix of the diamictite. When well developed, the fine-grained bornite is recognised visually through a significant darkening of the diamictite matrix. Chalcocite almost always occurs as fine-grained disseminations, particularly within the intermediate siltstone (Ki1.1.1.2). Supergene zones, in close proximity to dome edges, are typically chalcocite-dominant.

Figure 7.29 Strain-Shadow in DKMC_DD909

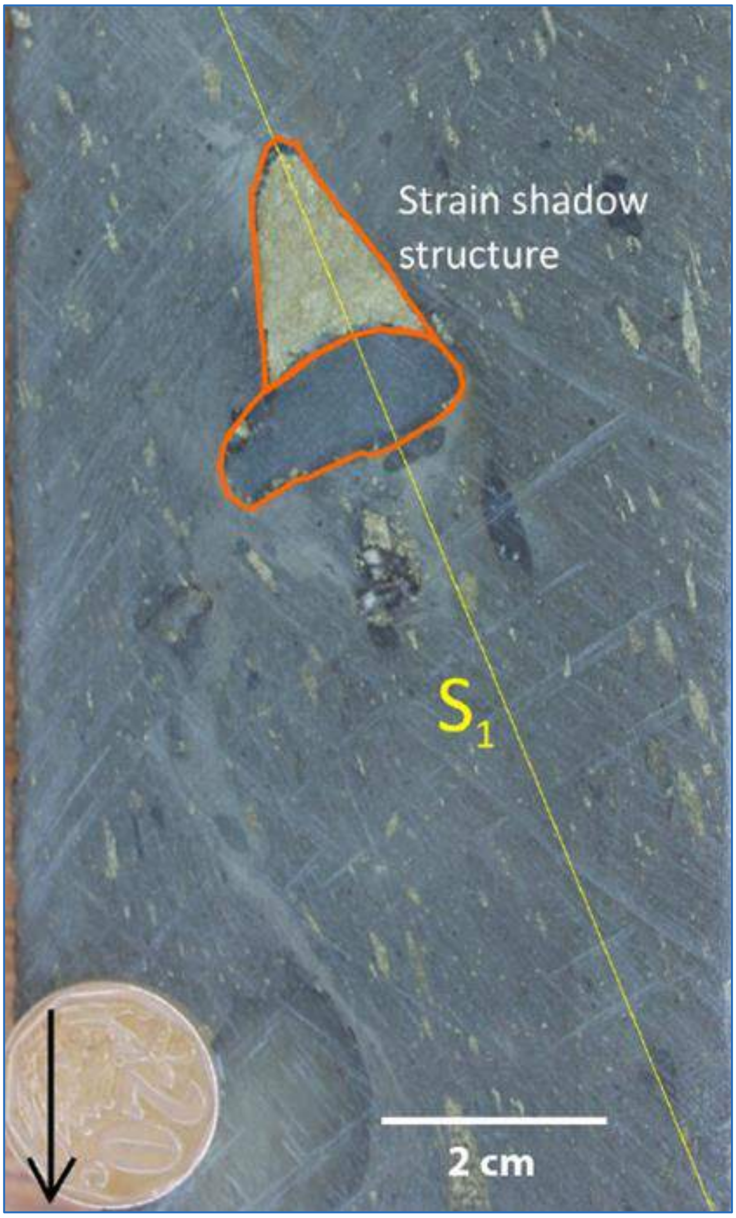


Figure provided by Ivanhoe, 2016. S_1 defines the orientation of the steep foliation.

Relative to Kamoa, the Kakula deposit is very different in its style of mineralisation. Whilst the vertical hypogene zonation is still evident at Kakula, the chalcopyrite and bornite zones are very narrow, with a very gradual transition downward from bornite to chalcocite, followed by a zone (typically within the basal siltstone) that is chalcocite-dominant (Figure 7.34). Whilst still dominantly fine-grained, numerous examples of coarse to massive chalcocite are evident in the highest-grade intersections (Figure 7.30).

Figure 7.30 Examples of Coarse to Massive Chalcocite at Kakula

DKMC_DD1055 at 347_m

DKMC_DD996 at 376_m

Figures provided by Ivanhoe, 2016.

Based on molar mass ratios, the theoretical TCu:S ratio (total copper divided by sulphur) for chalcopyrite, bornite and chalcocite was calculated and used to guide the identification of a change in sulphide species within individual drillholes. An overlap between sulphide species is generally observed in core and in thin sections; however, investigation of the TCu:S ratio shows that changes from one sulphide species to another can happen over distances of <1 m (Figure 7.31).

From the TCu:S ratio it is evident that the zonation develops regardless of the changes in copper grade. This is evident in DKMC_DD963, where the change to a theoretical chalcocite TCu:S occurs in the Ki1.1.1.1 even though the TCu grade has dropped below 1% (Figure 7.31). The full vertical zonation is not always developed, with chalcocite often being weakly developed to absent. The use of TCu:S ratios becomes unstable when TCu grades are <0.1% or sulphur values are <0.04%.

A scatter plot of TCu and S for all samples within the mineralised zone (Figure 7.32) shows a clear alignment along the different sulphide species, with a degree of scatter between these relating to transitional zones.

In contrast to Kamoa, mineralisation at Kakula is characterised by being chalcocite dominant, with gradual transition upward to bornite zones (Figure 7.32 and Figure 7.34). Chalcopyrite is observed in the core, but typically occurs outside of the defined mineralised zone, except in peripheral areas at Kakula West where the overall mineralised zone has narrowed, incorporating the full zonation.

Figure 7.31 Examples from Three Drillholes from Kamoa of Vertical Mineral Zonation Evident Based on TCu:S Ratios

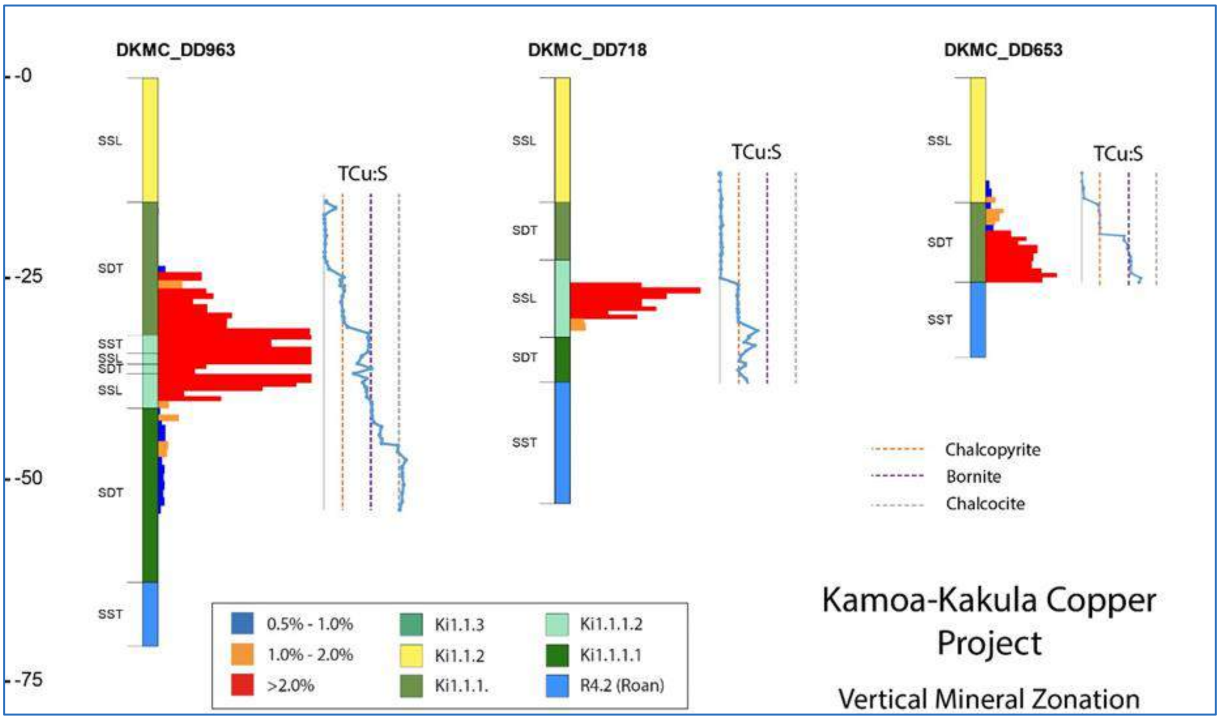


Figure provided by Ivanhoe, 2018.

Figure 7.32 Examples from Three Drillholes from Kakula of Vertical Mineral Zonation Evident Based on TCu:S Ratios

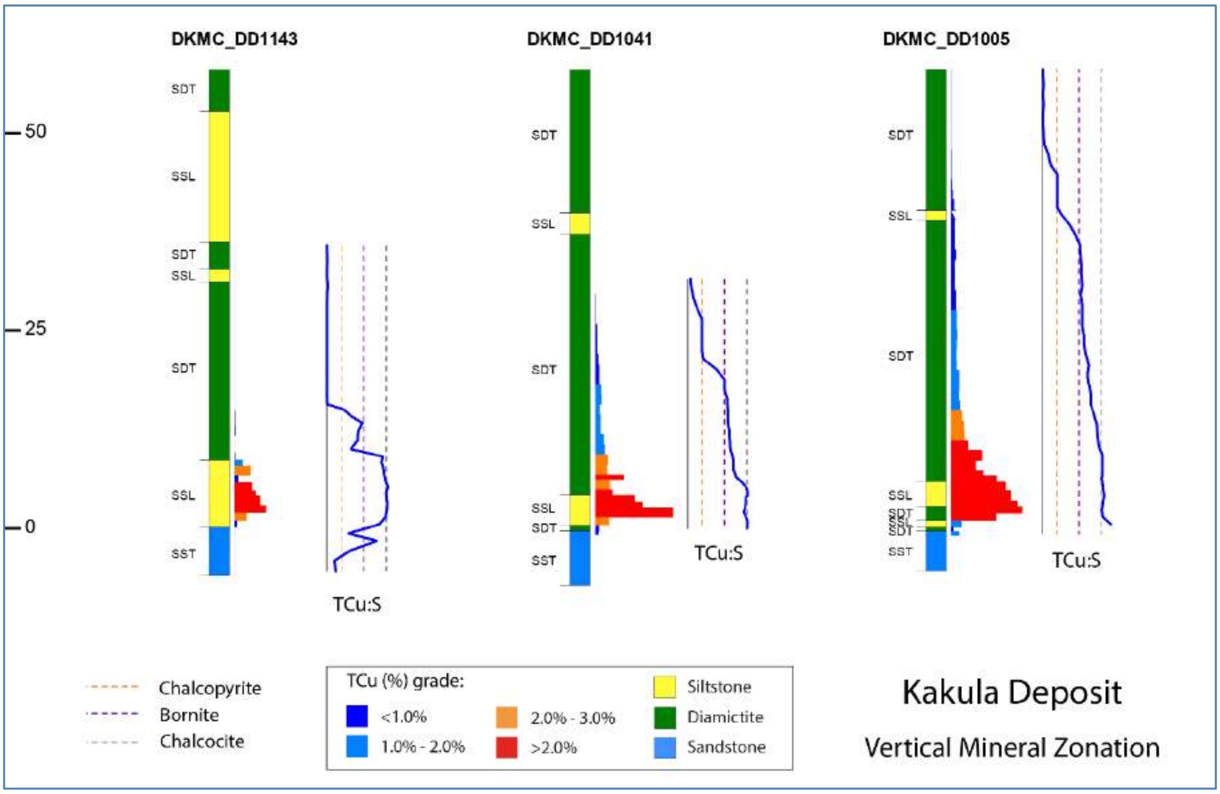


Figure provided by Ivanhoe, 2018.

Figure 7.33 Scatter Plot Illustrating Copper Sulphide Species Within the Mineralised Zone at Kamoa. Theoretical TCu: S Ratios for Chalcopyrite (orange), Bornite (purple) and Chalcocite (grey) are Based upon Molar Mass Ratios

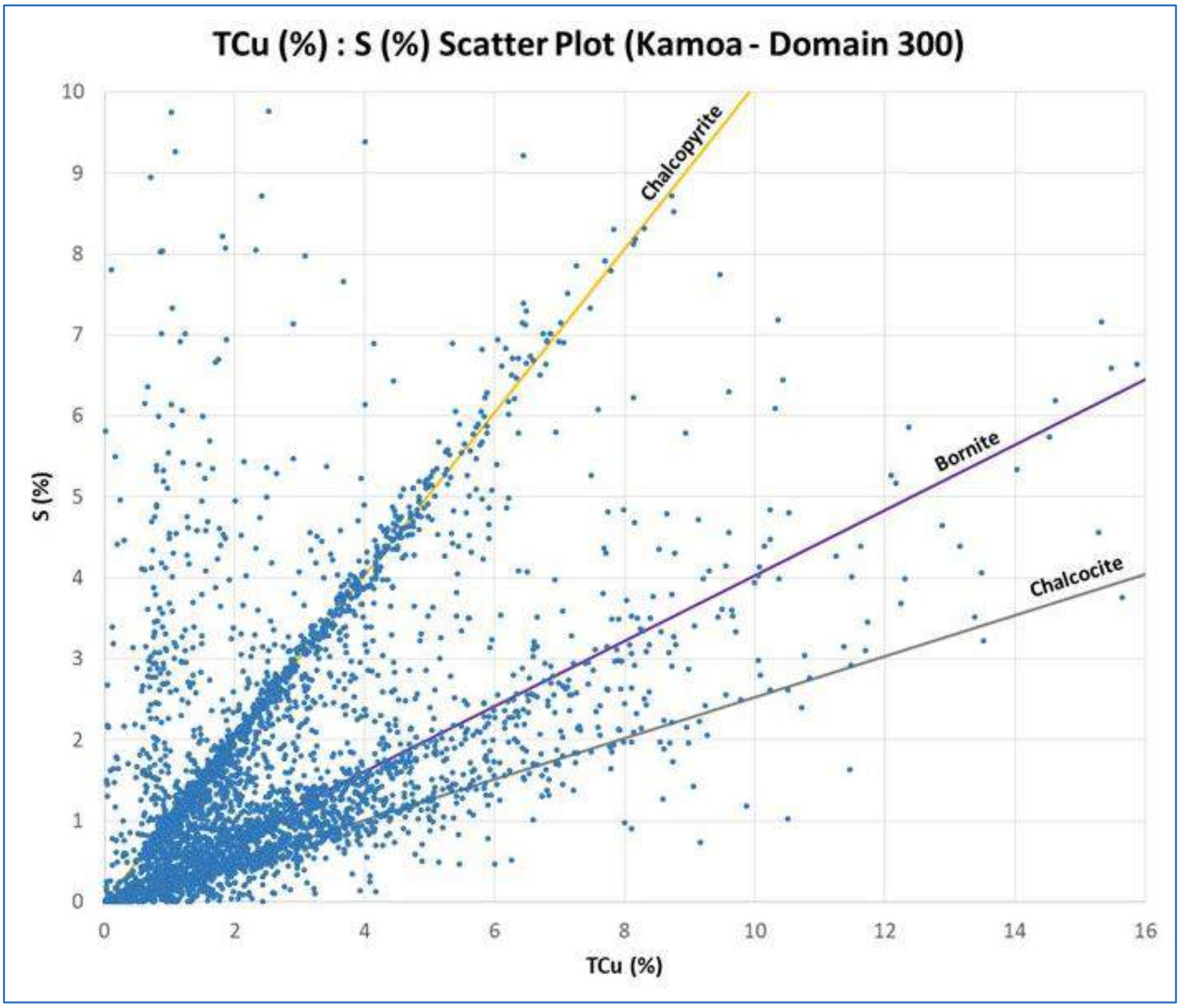


Figure provided by Ivanhoe, 2017. Points to left of the chalcopyrite line represent mixtures of pyrite and chalcopyrite; points to right and below the chalcocite line represent supergene enrichment (limited).

Figure 7.34 Scatter Plot Illustrating Copper Sulphide Species Within the Mineralised Domains (Domains 480 and 500) at Kakula. Theoretical TCu: S Ratios for Chalcopyrite (orange), Bornite (purple) and Chalcocite (grey) Based Upon Molar Mass Ratios

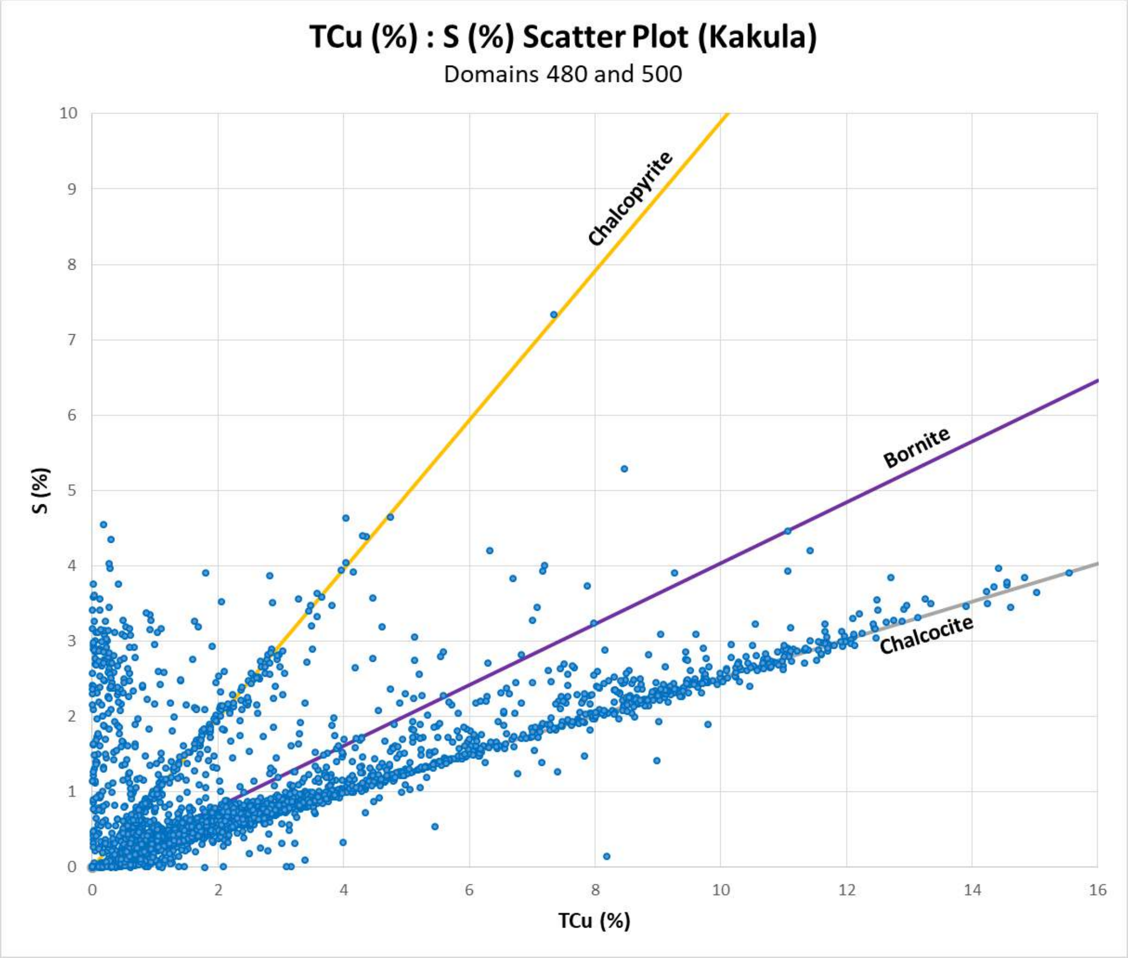


Figure provided by Ivanhoe, 2019. Points to left of the chalcopyrite line represent mixtures of pyrite and chalcopyrite; points to right and below the chalcocite line represent possible supergene enrichment (limited).

Stratigraphic Position of Mineralisation

The vertical position of mineralisation relates to the location of the reductant/s and proximity to the Roan aquifer. Although broadly stratiform, mineralisation does transgress stratigraphy when a lower reductant narrows or pinches out. Mineralisation is strongest, and the bottom-loaded profile is best developed, when the reductant is in direct, or very close contact, to the Roan aquifer. The mineralisation is not erratically developed in various stratigraphic positions; its position moves consistently and predictably from one unit to another (Figure 7.35).

Figure 7.35 Stratigraphic Section Showing Continuity of Mineralisation Near Base of Ki 1.1.1.3 at the Kamoa Deposit (8807500N looking North)

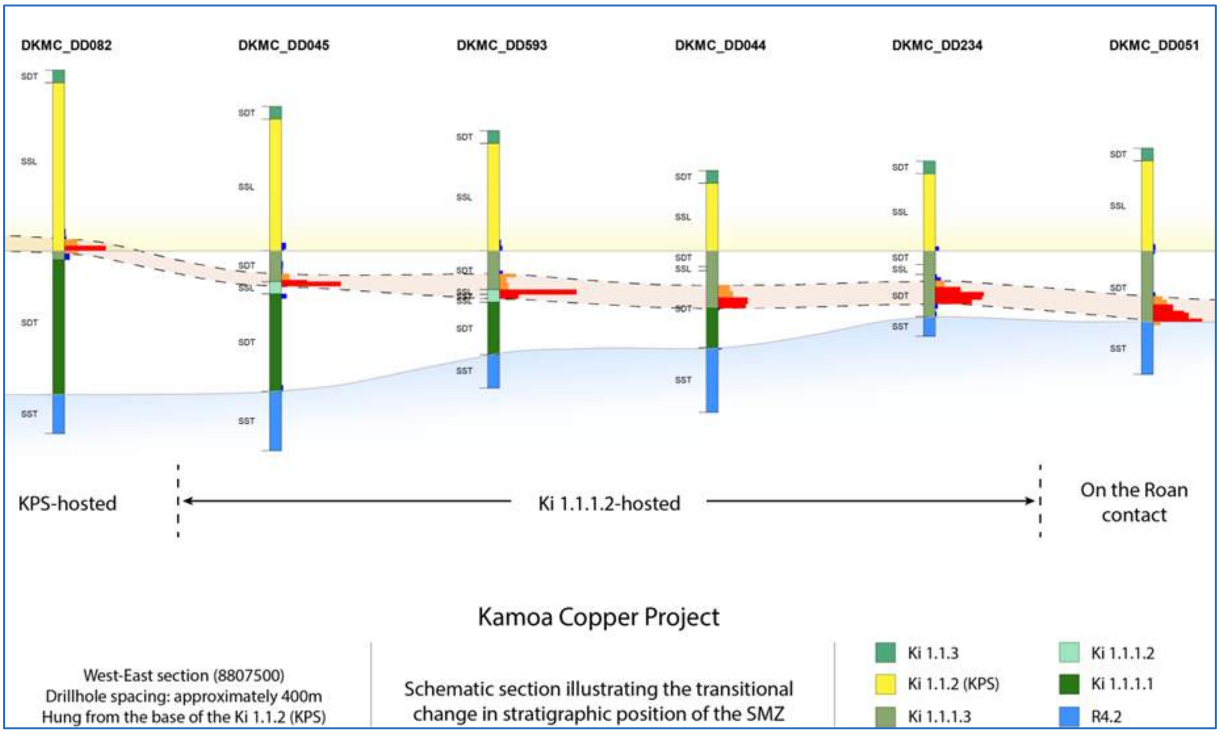


Figure provided by Ivanhoe, 2014. Copper grades in percent, shown as red histograms if over 1% TCu.

The nature of the copper grade distribution is related to its stratigraphic position and the localised development of lithological units. Where the mineralisation is located on the Roan contact, the mineralised interval is thick, and has a very strongly-developed bottom-loaded profile. Where the mineralisation is located at the base of the clast-poor diamictite (Ki1.1.1.3), the profile is typically bottom-loaded (if no intermediate siltstone is developed), or complex if one or more siltstone layers are developed. In the Kansoko Sud and Makalu areas, numerous siltstone layers developed within the diamictite cause the grade profile to become bimodal or even top-loaded. Where the mineralisation is hosted at the base of the KPS, it is typically narrow (but often high-grade), with a middle-loaded profile. The stratigraphic position of the mineralisation has been identified across the Project (Figure 7.36).

Figure 7.36 Facies in Which Mineralisation Occurs

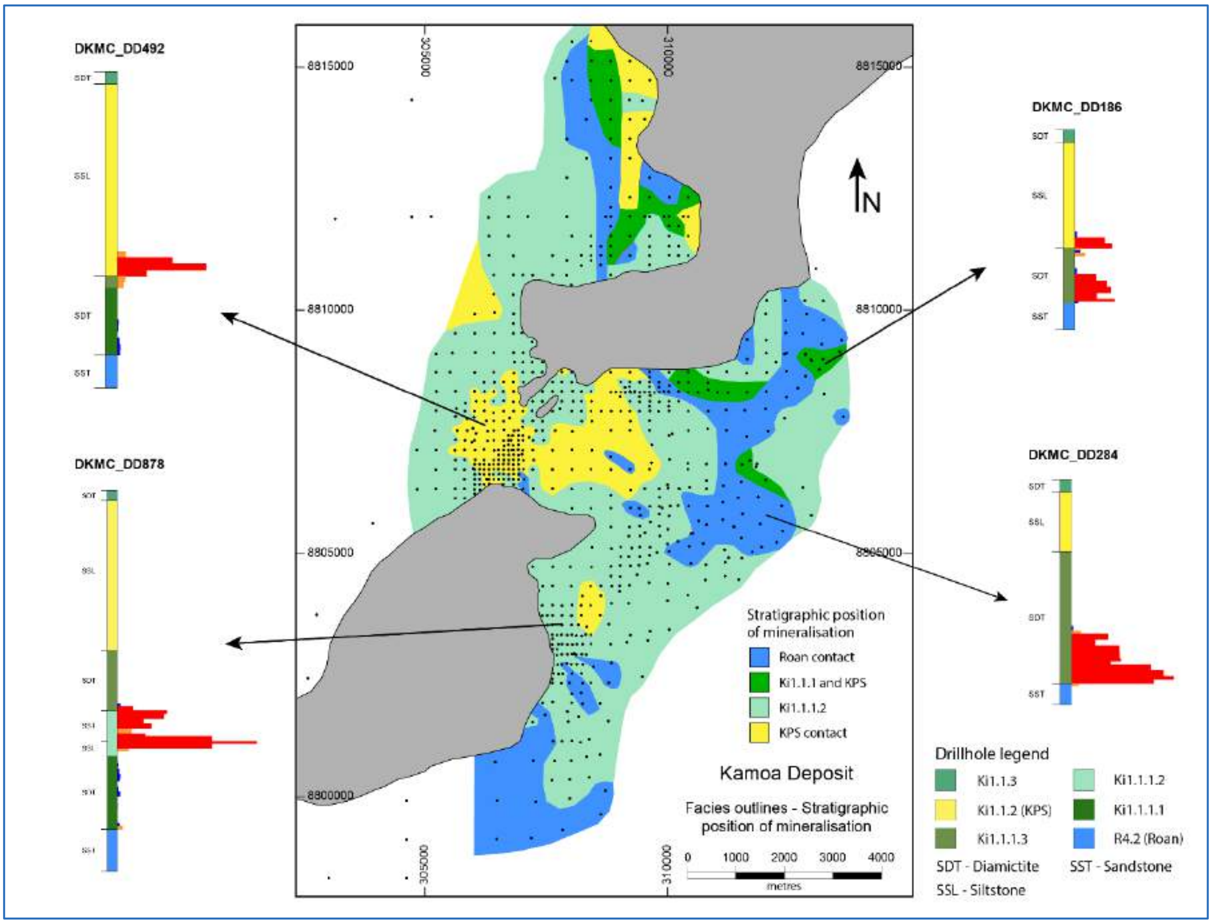


Figure provided by Ivanhoe, 2016. Copper grades in percent, shown as red histograms if over 1% TCu.

At Kakula, the mineralisation is consistently located just above (<5 m) the Roan contact. Immediately above the Roan contact, a narrow, weakly to poorly-mineralised clast-rich diamictite is developed, above which the basal siltstone occurs. Where this siltstone is present, it forms a very sharp contact with the highest-grade intersections. Grade profiles are almost always strongly bottom-loaded, allowing for well developed lateral continuity at higher cut-offs (3% TCu; Figure 7.37). In their shape, they resemble those at Kansoko Centrale, where the mineralisation occurs directly on the Roan contact; however, the Kakula grade profiles are usually considerably thicker and higher-grade (Figure 7.38).

Figure 7.37 Plan Image Illustrating the Continuity of High-grades due to the Bottom-Loaded Nature of the Mineralised Zone at Kakula

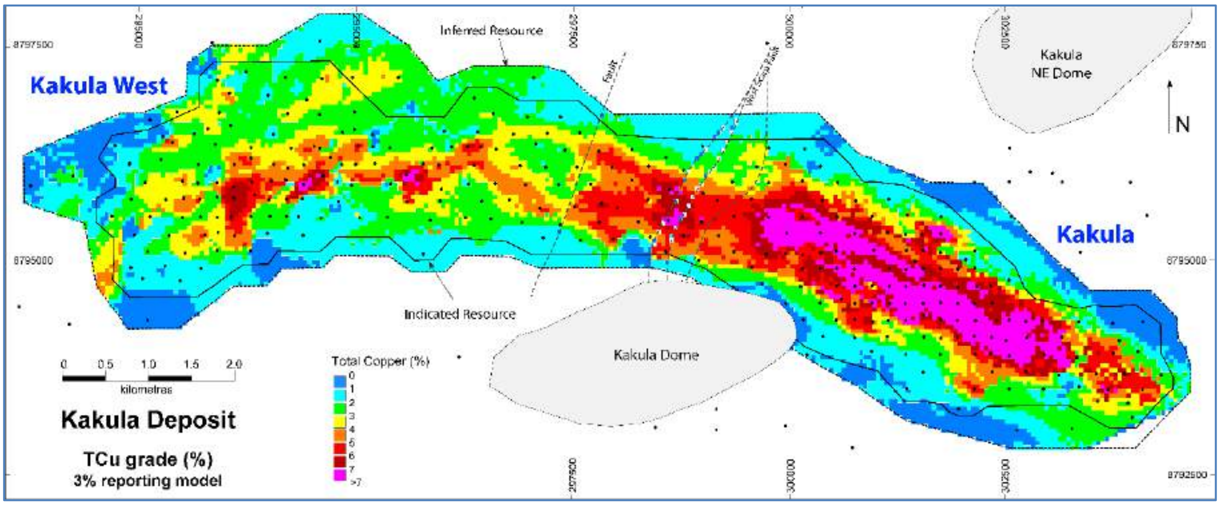


Figure provided by Ivanhoe, 2019. Missing blocks are due to fault offsets.

Figure 7.38 The Impact of Lithology on the Characteristics of the Grade Profile

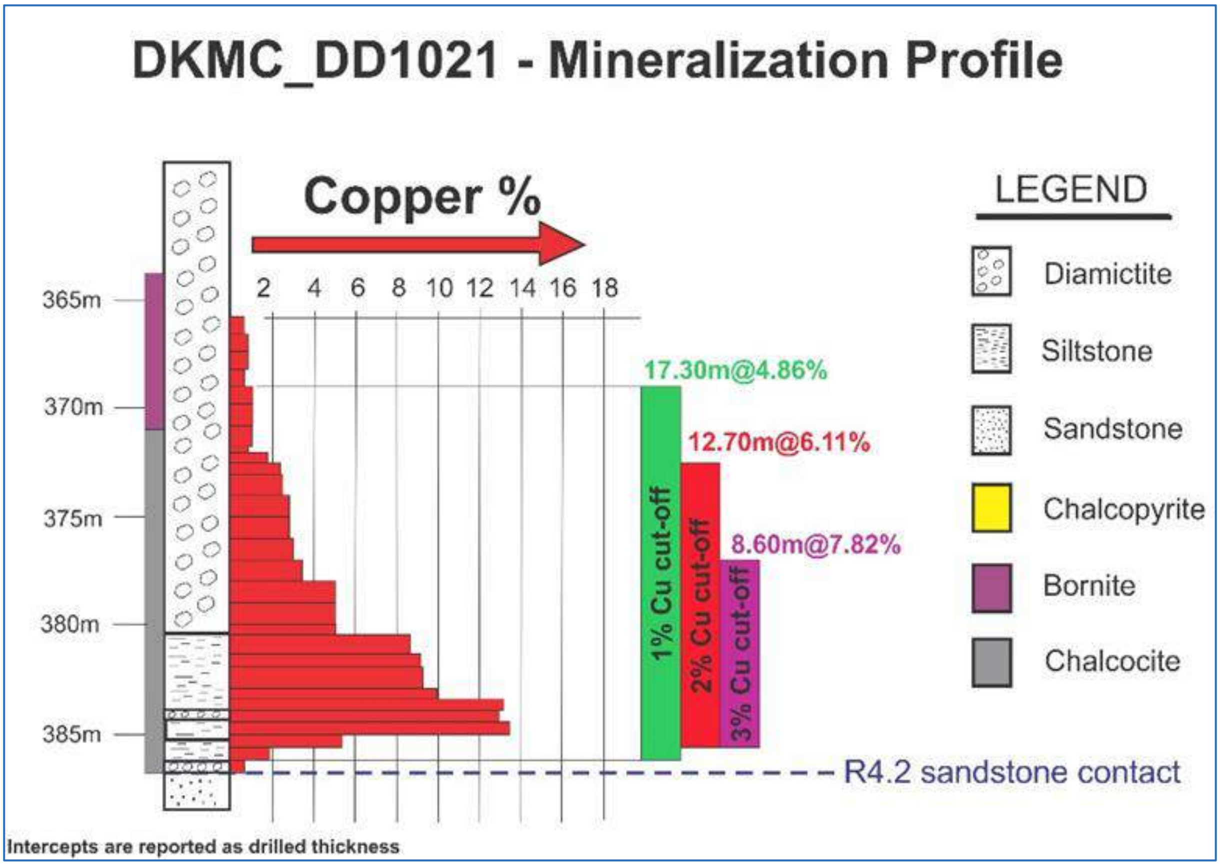


Figure provided by Ivanhoe, 2016.

In the central to south-east high-grade portions of Kakula, a strong relationship exists between the highest-grade intersections and the presence of the basal siltstone. Mineralised zones are thickest where the siltstone is thickest, and even within the gradually-weakening vertical grade profile, a sharp drop in grade can be observed at the top contact of the siltstone. A TCu (%) histogram of 1 m composite samples within the 3% grade shell shows a clear bimodal distribution. If the samples are separated based on a simplified host lithology (siltstone or diamictite), it becomes obvious that samples hosted in the siltstone account for the high-grade population, whilst the diamictite-hosted samples account for the lower-grade population (Figure 7.39). This relationship is considered to represent the distribution of the pyrite reductant prior to mineralisation, and has been incorporated into the domaining used in the estimation for both Kakula and Kakula West.

Figure 7.39 The Bimodal TCu (%) Distribution is Easily Explained by the Distinction Between Host Lithologies at Kakula

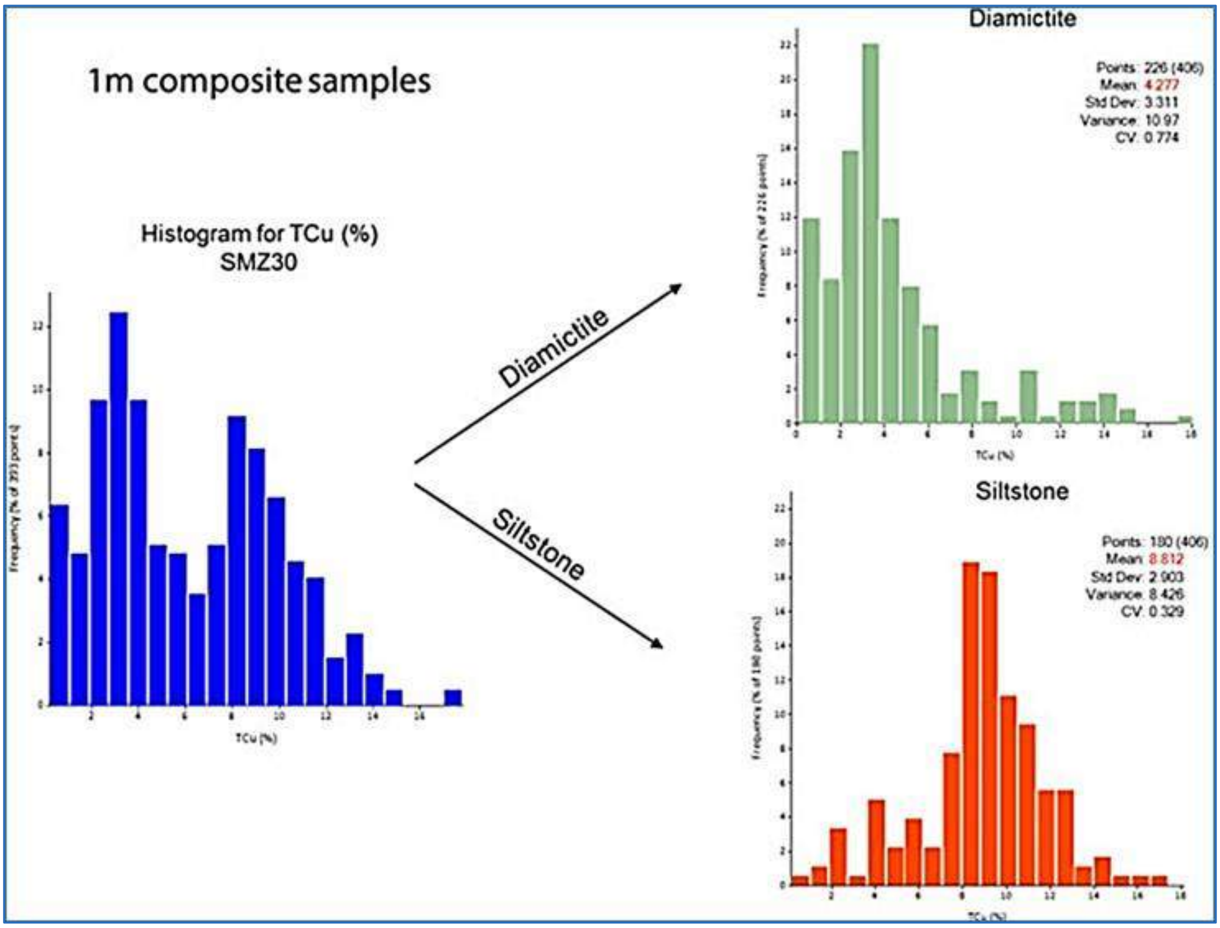


Figure provided by Ivanhoe, 2016.

A distinct maroon colour to the matrix of the diamictite has been observed in high-grade intersections. Investigations are still ongoing to explain this colour change, but it appears to highlight a control on the chalcocite mineralisation (Figure 7.40).

In the south-eastern portions of Kakula, the highest-grade intersections align very strongly along the 115° trend evident in the different stratigraphic and lithological units (refer to discussion in Section 7.3.1 and 7.3.2). To the north-west, the mineralisation turns to the west, with alignment along 105°. At Kakula West, well developed growth faults control the alignment of thickness and grade trends along variable north-east orientations. The intensity of these controls and their incorporation into the grade estimation are discussed in Section 14.

Figure 7.40 A Typical Basal, High-grade Portion of a Kakula Intersection, Highlighting the Maroon Colour and Basal Siltstone

Figure provided by Ivanhoe, 2016.

Mosaic Patterns to Mineralisation

Foreland-hosted copper deposits such as the Kamoa deposit show mosaic-patterns in terms of grade, thickness and stratigraphic position. In other words, detailed drilling (spacing 100 m or less) will often show areas that can be on the order of a kilometre in extent that have similar grade, thickness and stratigraphic position. These are termed mosaic pieces. At their edges, there can be significant changes to grade, thickness or stratigraphic position over a few hundred metres.

Figure 7.41 shows the November 2017 Kamoa Mineral Resource model for TCu and true thickness with superimposed drillholes. There are clear discontinuities in grade and thickness around mosaic pieces running greater than 3.5% TCu or having true thicknesses over 10 m.

Figure 7.41 TCu Grade (left) and Vertical Thickness (right) for the Kamoa Deposit 2017 Mineral Resource

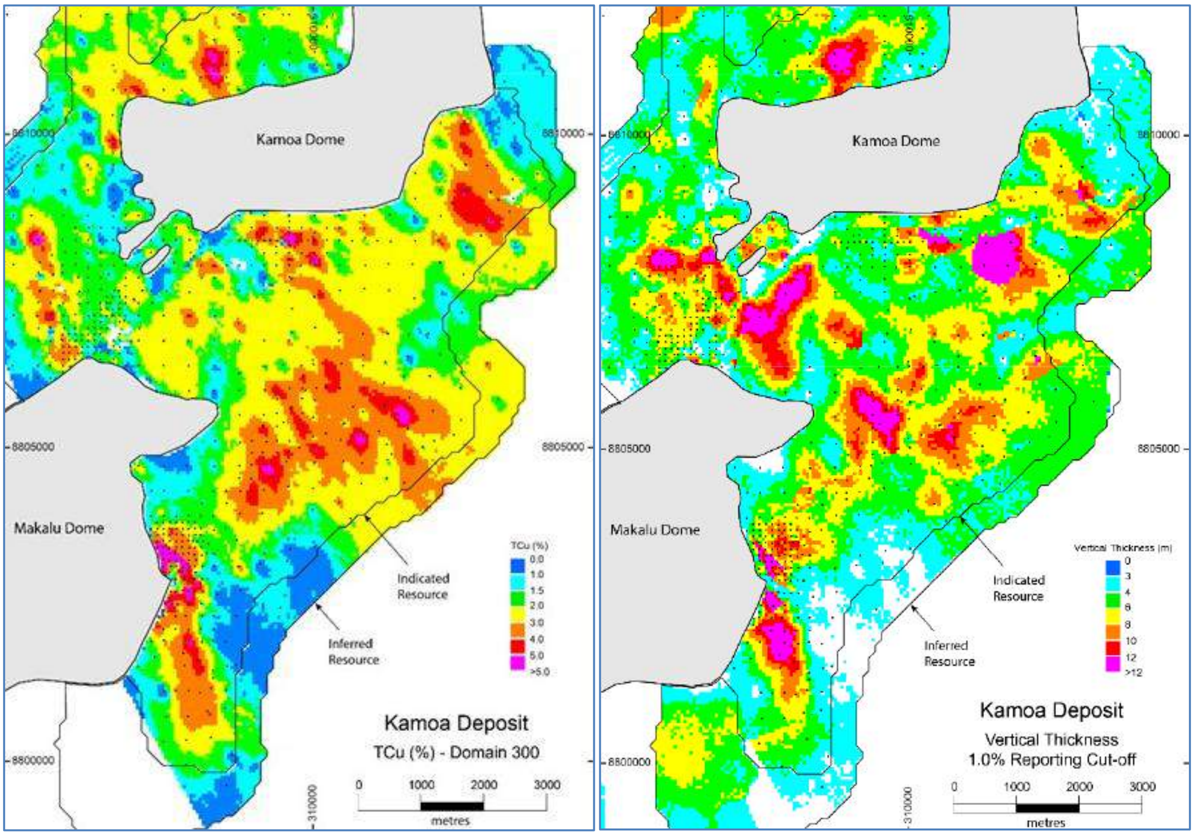
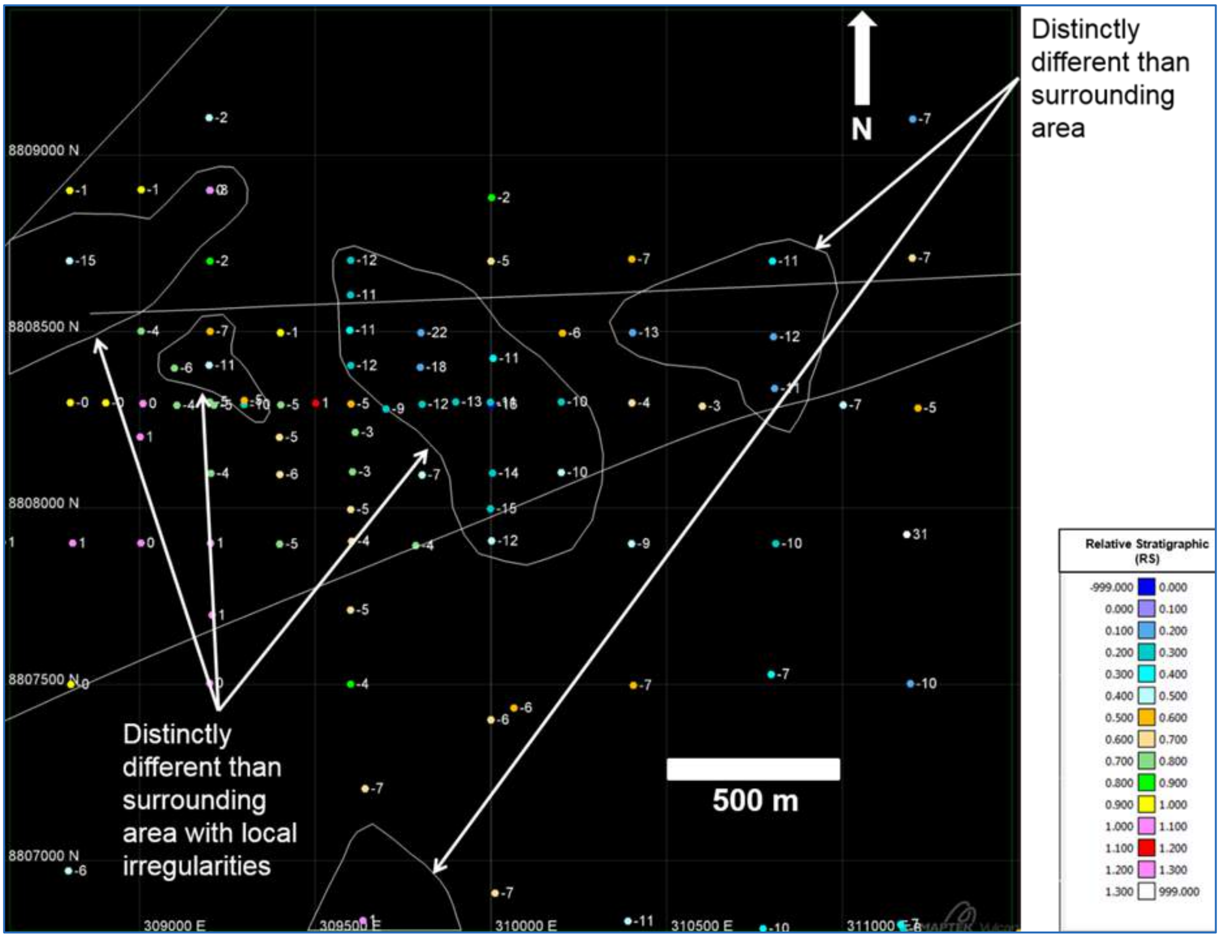


Figure provided by Ivanhoe, 2018. Average grade for blocks from the 3D model selected within the 2D SMZ10 wireframe for Domain 300 (Upper SMZ). Vertical thickness from the 3D model selected above a 1% reporting cut-off.

Figure 7.42 shows an area at the Kamoa deposit delineated using 200 m spaced drillholes. Plotted is the distance between the base of the KPS and the centroid of the SMZ10, the mineralised intercept at a 1%, TCu cut-off over 3 m used to select the mineralised zone (SMZ10). The stratigraphic position of the SMZ in relation to the bottom of the KPS unit and top of the Roan unit was reviewed by calculating the relative stratigraphic (RS) position {(RS=1- [(KPSz – SMZz)/KPSz – ROANz)]}. Again, discontinuities are present at the edges of mosaic pieces.

Figure 7.42 Stratigraphic Position of SMZ10 with Respect to the Base of the KPS



Source (Seibel 2014): drillholes are colour coded by relative stratigraphic position; posted values are actual elevation differences in metres between centroid of SMZ10 and base of KPS; negative numbers indicate the centroid of SMZ10 is below the base of the KPS.

The shapes of the mosaic pieces are irregular, and the non-linearity of the edges does not support an explanation by faulting, but rather may reflect the eH–pH conditions at the time of deposition of the mineralisation and/or pre-mineralisation sulphide concentration in the diamictite.

7.4 Comments on Section 7

The Amec Foster Wheeler QPs note the following:

- The understanding of the deposit settings, lithologies, and geological, structural, sulphide mineralogical, and alteration controls on mineralisation is sufficient to support estimation of Mineral Resources and Mineral Reserves at Kamoa, and Mineral Resources at Kakula.
- Mineralisation within the Project has been defined over an irregularly-shaped area of 27 km x 21 km. The mineralisation is typically stratiform, and vertically zoned. The dip of the mineralised body ranges from 0° to 10°, to 15° to 20° on the flanks of the dome.
- The occurrence of copper mineralisation in mosaic pieces was also seen by Dr. Parker in the 1990s, from the results of underground drilling at Konkola, Zambia.
- Definition of the edges of the mosaic pieces will require close-spaced drilling on the order of 50 m or less.
- Typically, contaminants are not a problem for Copperbelt-style deposits. The initial 2010 drilling programme at Kamoa had assayed for a large number of potential contaminants, including As, Zn, Pb, Mn, and Fe. Increased concentrations of As (typically 50 to 150 ppm) and Zn (0.1 to 0.5%) were found in local areas where the copper mineralisation occurs near the contact with the KPS. Assaying for these elements was discontinued by Ivanhoe in 2010–2011 after Amec Foster Wheeler (Reid, 2010a) showed a good correlation between minor element assays with Niton (X-ray fluorescence or XRF) results, and the Niton results are adequate to identify any areas where contaminants may be of concern.

8 DEPOSIT TYPES

The mineralisation identified to date within the Project is typical of sediment-hosted stratiform copper deposits. Such deposits can be hosted in either marine or continental (red-bed) sediments. Major global examples of these deposits include the Kupferschiefer (Poland), most of the deposits within the Central African Copperbelt (such as Konkola, Nkana, Nchanga, Mufulira, Tenke–Fungurume, and Kolwezi), Redstone (Canada), and White Pine (USA).

Common features of sediment-hosted copper deposits are (Kirkham, 1989; Hitzman et al., 2005):

- Geological setting: Intracratonic rift; fault-bounded graben/trough, or basin margin, or epicontinental shallow-marine basin near paleo-equator; partly evaporitic on the flanks of basement highs; sabkha terrains; basal sediments highly permeable. Sediment-hosted stratiform copper deposits occur in rocks ranging in age from Early Proterozoic to late Tertiary, but predominate in late Mesoproterozoic to late Neoproterozoic and late Palaeozoic rocks.
- Deposit types:
  - Kupferschiefer-type: Host rocks are reduced facies and may include siltstone, shale, sandstone, and dolomite; these rocks typically overlie oxidised sequences of haematite-bearing, coarser-grained, continental siliciclastic sedimentary rocks (red beds). As the host rocks were typically deposited during transgression over the red bed sequence, these deposits tend to have exceptional lateral extents. The Central African Copperbelt deposits are typical of the Kupferschiefer type.
  - Red-bed-type: Isolated non-red rocks within continental red-bed sequences. Occur typically at the interface between red (haematite-bearing) and grey (relatively reduced, commonly pyrite-bearing) sandstone, arkose, or conglomerate. The configuration of the mineralised zone varies from sheet-like, with extensive horizontal dimensions, to tabular or roll-front geometries, with limited horizontal dimensions.
- Mineralisation: Deposits consist of relatively thin (generally <30 m and commonly less than 3 m) sulphide-bearing zones, typically consisting of haematite–chalcocite–bornite–chalcopyrite–pyrite. Some native copper is also present in zones of supergene enrichment. Galena and sphalerite may occur with chalcopyrite or between the chalcopyrite and pyrite zones. Minerals are finely disseminated, stratabound, and locally stratiform. Framboidal or colloform pyrite is common. Copper minerals typically replace pyrite and cluster around carbonaceous clots or fragments.
- Mineralisation timing: Sulphides and associated non-sulphide minerals of the host rocks in all deposits display textures and fabrics indicating that all were precipitated after host rock deposition. Timing of mineralisation relative to the timing of host-rock deposition is variable, and may take place relatively early in the diagenetic history of the host sediments or may range to very late in the diagenetic or post diagenetic history of the sedimentary rocks.
- Transport/pathway: Porosity in clastic rocks, upward and lateral fluid migration; marginal basin faults may be important; low-temperature brines; metal–chloride complexes.

- Metal deposition: Metals were characteristically deposited at redox boundaries where oxic, evaporite-derived brines containing metals extracted from red-bed aquifers encountered reducing conditions.
- Mineralisation controls: Reducing low pH environment such as marine black shale; fossil wood, and algal mats are important as well as abundant biogenic sulphides and pyritic sediments. High permeability of footwall sediments is critical. Boundaries between hydrocarbon fluids or other reduced fluids and oxidised fluids in permeable sediments are common sites of deposition.
- Alteration: Metamorphosed red-beds may have a purple or violet colour caused by finely-disseminated haematite.

8.1 Comments on Section 8

Many features of the mineralisation identified within the Project to date are analogous with the Polish Kupferschiefer-type deposits and the stratabound, sediment-hosted, Zambian Ore Shale deposits, in particular the Konkola, Nchanga, Nkana, and Luanshya deposits.

Key features of the deposits include:

- Laterally continuous, have been drill tested over an area of 27 km x 21 km.
- Strong host-rock control and restriction of the mineralisation to a redox boundary zone between oxidised footwall haematitic sandstone and reduced, sulphidic host diamictites and siltstone-sandstone rocks.
- Presence of the replacement, blebby, and matrix textures that are typical of sediment-hosted copper deposits.
- Vertical zoning of disseminated copper sulphide minerals from chalcocite to bornite to chalcopyrite.
- Hypogene minerals are chalcopyrite, bornite and chalcocite, with the predominant copper sulphide species varying spatially throughout the deposit. For example, deep drilling along the Kansoko Trend has intersected mixtures of bornite and chalcocite. Mineralisation at Kakula is predominately chalcocite.
- Occurrence of very fine-grained, bedded, disseminated copper sulphides in the intermediate sandy siltstone unit (Ki1.1.1.2) within the basal diamictite, or within the basal siltstone at Kakula, is typical of Zambian Ore Shale-style mineralisation.

The virtual absence of carbonate rocks and the absence of widespread silicification both as host-rock alteration and in veins is atypical of the Mines Subgroup-hosted deposits of the Katangan Copperbelt (e.g. Tenke–Fungurume). Locally minor dolomite replacement of sulphidic clast rims in the basal diamictite and scattered tiny carbonate +/- quartz veinlets with occasional sulphides can occur at the Kamoa deposit.

Exploration programs that use a stratabound, sediment-hosted model are considered applicable to the Project area.

9 EXPLORATION

Prior to commencement of on-ground exploration in 2004, Ivanhoe commissioned data acquisition in 2003 by African Mining Consultants and The Mineral Corporation. This work comprised collation of the following:

- Landsat 7 ETM+ imagery.
- Shuttle Radar Topographic Mission (SRTM) digital elevation model.
- Geological maps (1:20,000 to 1: 2,000,000 scale; Francois (1996) and (1997)).
- Mineral occurrence maps.
- Russian topographic maps for the Katanga Province (1: 250,000 scale).

The collated data were used to identify areas that were considered more prospective within Ivanhoe's then tenure holdings.

9.1 Grids and Surveys

All surveys to date are in UTM co-ordinates, using the WGS84 projection, Zone 35S.

In 2004, a topographic survey, as part of the airborne magnetic-radiometric survey was flown over the Project, resulting in production of a topographic contour map that is accurate to 12 m. Ivanhoe obtained higher resolution, light detection and ranging (LiDAR) based, topographic data over the Project area in 2012.

9.2 Geological Mapping

Project mapping has been performed at 1:150,000, 1:100,000, and 1:5,000 scales where outcrop permits. Over most of the Project area, there is little or no significant geological exposure. There is one small outcrop on the Kamoa dome, identified by Ivanhoe to be a basal conglomerate of the Lower Roan Poudingue (below the R4.2), located near its unconformable contact with Kibaran quartzite below.

A reconnaissance field mapping programme occurred between August and October 2010 at the Kakula deposit. The purpose of this programme was to delineate the edge of the sandstone dome and its contact with the overlying diamictite known to crop out in this area. The contact formed a Kamoa-style target type, and previous surface geochemical programmes have delineated elevated copper associated with this contact. The mapping successfully delineated the contact, and drilling of the Kakula deposit commenced in 2015.

9.3 Geochemical Sampling

Geochemical and aircore drill sampling programmes were conducted as part of first pass exploration and used to create vectors into mineralisation. Geochemical sampling programmes included stream sediment, soil and termite mound sampling.

9.4 Geophysics

During 2004, a regional airborne geophysical survey was flown by Fugro Airborne Surveys (Pty.) Ltd. on behalf of Ivanhoe. Data processing was completed using Oasis Montaj software from Geosoft Inc. of Toronto, Canada. The programme identified a number of magnetic lineaments that reflect underlying structures. One major structural set is interpreted to be a suture zone between the thrust and fold belt to the east and stable Proterozoic sediments that have been draped over domes and fill broad basins in the Project area. A second structural set relates to normal, post-mineralisation faults, which appear to have large displacements.

In 2011, Gap Geophysics Australia and Quik_Log Geophysics conducted downhole electromagnetics (EM) surveys on three holes at Kamoa, DKMC_DD322, DKMC_DD325, and DKMC_DD330. The data collected included natural gamma, density, sonic, magnetic susceptibility, three component magnetics, resistivity, conductivity, induced polarisation and acoustic data (fractures). This was intended to be an orientation study on the effectiveness of using downhole geophysics as a future tool and determination of which instruments/tests provide useful information. Preliminary results suggested that the televiewer may be a useful tool in conjunction with the geotechnical logging.

As well, in 2011, an EM orientation survey line was completed to test the effectiveness of EM as a possible exploration tool. The line was completed east–west across Kansoko coincident with one drill section line to facilitate comparison with the geologic record logged in drillholes. Results were inconclusive.

A ground magnetic survey was completed over the Kamoa deposit area during 2011–2012. Instruments purchased by Ivanhoe were used by locally-employed teams. The ground magnetic data have been compiled to help with geology and structure mapping.

Ground gravity data were collected from eight lines at Kakula in 2016 to aid in mapping of the Ki1.1.1–R4.2 contact. These data were calibrated against 7,327 core density measurements and six wireline logs and combined with the geology model for Kakula. A 3,100 km airborne gravity survey was conducted over 2,000 km^2 of the Western Foreland area (including Kamoa-Kakula) in January 2018. Information from this survey will be used for exploration vectoring.

In 2016 and 2017, Quick Log Geophysics conducted downhole surveys on 12 drillholes at Kakula. The data collected included logged full wave sonic, dual density, resistivity and gamma. Acoustic televiewer (ATV) data were also obtained for use in ongoing geotechnical work.

In September 2017, HiSeis commenced acquisition of four regional scale 2D seismic lines; two north–south lines, and two west–east lines, using a vibrating rig and traditional inline seismic sensors linked to a monitoring station. The survey aimed to position the top of the Roan, interpret broad-scale basin architecture and locate both known and unknown growth and younger brittle structures. The survey was completed in June 2018 following a delay during the wet season. Vertical seismic profiles (VSPs) were completed on seven holes along these lines. The results are being used to aid depth conversion of the seismic data and interpretation of seismic signals.

Integration of the geophysical programme results with the Kamoa-Kakula team's existing geological models will allow fine-tuning of exploration targeting within the highly prospective Kamoa-Kakula exploitation licence area.

9.5 Petrology, Mineralogy, and Research Studies

Whole-rock major and trace element data were collected by Ivanhoe in 2009 from the mineralised zone and footwall sandstone in drillhole DKMC_DD019. Analyses were completed at Ultra Trace laboratories, and included a standard (10 element plus SO_3 and loss-on-ignition (LOI) X-ray fluorescence (XRF) major element suite, and a 46 element inductively coupled plasma (ICP) trace element suite. Results indicated possible K_2O enrichment commensurate with potassic (feldspar–sericite) alteration.

An MSc study was completed at the Colorado School of Mines on the stratigraphy, diagenetic and hydrothermal alteration, and mineralisation, and an accompanying paper has been published in *Economic Geology* (Schmandt, et al, 2013).

The main conclusions from the study are:

1. The Grand Conglomérat diamictite was formed by glacially-derived mass transport and sedimentary gravity flows in a tectonically active, locally anoxic marine environment.
2. The early diagenetic framboidal and later cubic pyrite associated with the copper mineralisation may be indicative of early hydrothermal activity.
3. Later hydrothermal alteration mineral assemblages within the lower Grand Conglomérat are stratigraphically zoned, trending from a potassic and silicification assemblage in the lowermost stratigraphic units to a dominantly magnesium alteration assemblage higher up in the stratigraphy.
4. Sulphur isotope studies indicate that most of the sulphur in the copper sulphides was derived from early diagenetic iron sulphide.
5. Fluid inclusion analyses indicate that the mineralisation-forming fluid was saline, ~23 to 26 wt% NaCl wt equivalent, and had homogenisation temperatures (Th) ranging from 210° to 240°C.

Two additional studies completed recently have been summarized in papers released in the journals Sedimentology (Kennedy et al., 2018) and African Journal of Earth Science (Twite et al., 2019). These studies highlighted the importance of syn-sedimentary growth faults and their role in localizing high-grades (Twite et al., 2019), and the origin of the thick diamictite packages as subaqueous debris flows (rather than primary glcial deposits) in response to faulting and rapid subsidence of the basin (Kennedy et al., 2018).

Ivanhoe, through the Laurentian–Ivanhoe Mines Education partnership is part-funding two PhD research projects and three MSc research projects on Kamoa-Kakula. Areas of research include:

- Mineralising fluids of the Kamoa-Kakula deposits;
- The Geologic History of the diamictite matrix at Kamoa-Kakula;
- U–Pb geochronology of the Kamoa-Kakula host succession;
- Stratigraphic and geochemical controls on Kamoa-Kakula;
- Re-Os geochronology of the Kamoa-Kakula ore minerals.

9.6 Exploration Potential

The Kamoa-Kakula Project area is underlain mainly by subcropping Grand Conglomerat diamictite, the base of which occurs at the Kamoa and Kakula deposits, and thus the entire area underlain by diamictite can be considered prospective for discovery of extensions to the known mineralisation, and for new zones of mineralisation within this same horizon. With more drilling, the exploration potential for expanding the area of known mineralisation that is hosted in diamictite is excellent.

Initial exploration programs identified a number of priority grass-roots exploration prospects within the Project, based on geological interpretations, stream-sediment and soil sampling, and aircore, RC, and core drilling. The most prospective area, Kakula, has been drill tested, modelled, and Mineral Resources have been estimated. A westerly extension to Kakula (Kaklua West) identified in 2017 has now also been drill tested, and is has Mineral Resources estimated. A target for further exploration (Kamoa-Makalu) is discussed in Section 14.17.

In addition, and by analogy with the Zambian and Katangan districts of the Central African Copperbelt, it is possible that multiple (stacked) redox horizons and associated stratiform copper zones exist within the Roan sequence, hidden below the diamictite. Because of the difficulty in detecting or predicting mineralisation below the diamictite footwall, Ivanhoe considers that the most reliable means of evaluating this hypothesis is with wide-spaced deep drillholes. This approach is predicated on the assumption that stratiform copper deposits are laterally extensive, and occur at the kilometre scale.

9.7 Comments on Section 9

In the opinion of the Amec Foster Wheeler QPs:

- The exploration programmes completed to date are appropriate to the style of the Kamoa and Kakula deposits.
- The research work that has been undertaken supports Ivanhoe’s genetic and affinity interpretations for the Project area.
- The Project area remains prospective for additional discoveries of base-metal mineralisation within diamictites around known dome complexes.
- Anomalies generated by geochemical and drill programmes to date support additional work on the Project area.

10 DRILLING

10.1 Introduction

The drillhole database used for the Kamoa resource estimation was closed on 23 November 2015. The Kakula resource model described in this Report, is separated into two parts, with the West Scarp Fault forming the boundary. The drillhole database for the Kakula model, east of the West Scarp Fault, is based on drilling up to 26 January 2018. The drillhole database for Kakula West, west of the West Scarp Fault, was closed on 1 November 2018.

Aircore, RC and core drilling have been undertaken since May 2006. Aircore and RC drilling were used in early exploration to follow up identified anomalies. None of these drillholes are used for resource estimation. Coreholes have been used for geological modelling, and those occurring within the mining lease and in areas of mineralisation (drillholes on the Kamoa, Makalu and Kakula domes are excluded) have been used for resource estimation.

As of 1 March 2019, there were 1,853 coreholes drilled within the broader Project area (Table 10.1). The statistics in Table 10.1 are based on the current drillhole collar data provided to Amec Foster Wheeler in the form of an Excel spreadsheet by Ivanhoe (Gilchrist, 2019). The 2017 Kamoa Mineral Resource estimate used 776 drillhole intercepts. Included in the 776 drillholes are 16 twin holes (where the spacing between drillholes is <25 m) and six wedge holes. Although a far greater number of holes have been wedged, the wedges have typically been used in their entirety for metallurgical testing, and have thus not been sampled for resource estimation purposes. In these cases, only the parent hole is used during Mineral Resource estimation.

The April 2018 Kakula Mineral Resource estimate used 155 drillhole intercepts collared east of the West Scarp Fault. The November 2018 Kakula West Mineral Resource estimate used 168 drillhole intercepts collared west of the West Scarp Fault. The 754 holes not included in either the Kamoa or Kakula estimates were excluded because they were either abandoned, unmineralised holes in the dome areas, unsampled metallurgical, civil geotechnical or hydrological drillholes, or were drilled after the closure of the database for the Kamoa Mineral Resource estimation (23 November 2015) or after the closure of the database for the Kakula Mineral Resource estimation (26 January 2018), or after the closure of the database for the Kakula West Mineral Resource estimation (1 November 2018) refer to Table 10.1.

Figure 10.1 shows the collar locations of drillholes occurring inside the Project area as of 1 March 2019. Figure 10.2 shows the completed drilling at Kakula as of 1 March 2019. Coreholes typically commence collecting cores at PQ size (85 mm), reducing to HQ size (63.5 mm), and where required by ground conditions, further reducing to NQ size (47.6 mm).

Table 10.1 Drilling Statistics per Drill Purpose for Coreholes (as of 1 March 2019)

Drill Purpose	**Count (Active)**	**Metres (m)**
	Resource	
Kamoa estimate (Nov 2017)	776	225,620.4
Kamoa (post-estimation)	110	30,366.0
Kakula estimate (April 2018)	155	70,778.50
Kakula (post-estimation)	20	11,426.2
Kakula West estimate (Nov 2018)	168	111,935.0
Kakula West (post-estimation)	28	20,115.2
Exploration	13	6,400.1
Domes	114	10,678.2
Metallurgy	131	15,433.0
Geotechnical	50	6,017.4
Civil Geotechnical	98	2,773.6
Condemnation	51	1,177.8
Cover Drilling	25	3,803.0
Permeability	7	60.4
Abandoned	107	22,958.5
Total	**1,853**	**539,543.3**

Note: Wedge holes are counted as individual drillholes in this table, although the drill meterage only includes the wedged portion of the drillhole. If a wedge hole used in the Mineral Resource estimate was wedged off an abandoned parent hole, the full meterage from surface is assigned to the resource category and only the residual portion assigned to 'Abandoned'. 'Exploration' holes refer to those holes outside of the modelled Mineral Resource area, or wedges drilled primarily for academic study. If a drillhole was drilled for geotechnical or metallurgical purposes but has been used in the Mineral Resource estimate, it is classified as a resource drillhole.

Figure 10.1 Mineral Resource Definition Drilling at Kamoa-Kakula

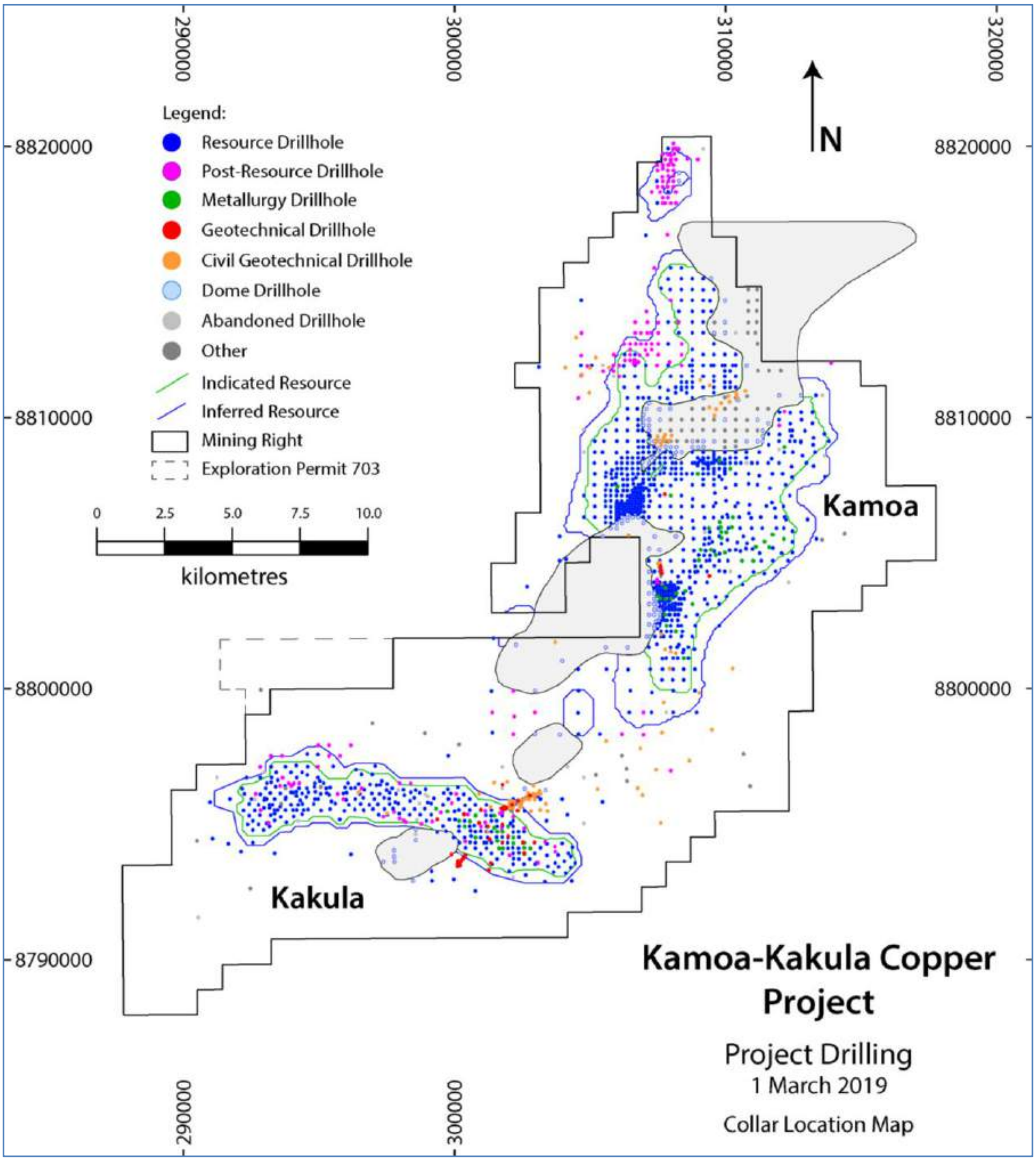


Figure provided by Ivanhoe, 2019. 'Other' includes exploration drillholes, condemnation drillholes, cover drillholes and permeability drillholes.

Figure 10.2 Drill Location Plan, Kakula

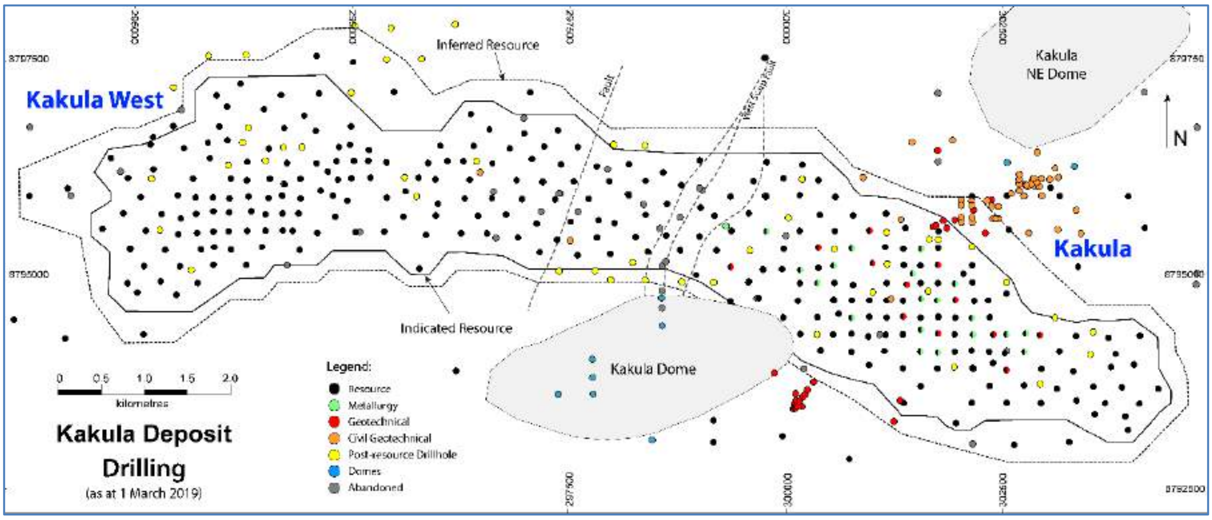


Figure provided by Ivanhoe, 2019. Positions shown are end-of-hole co-ordinate positions.

10.2 Geological Logging

Standard logging methods, sampling conventions, and geological codes have been established for the Project. Free-form description was allowed in the description section of the drill log where any unusual features worthy of description were noted. The geological logging for the minor stratigraphy at Kakula is still in development.

Drill core, RC, and aircore chips were logged by a geologist, using paper forms, which capture lithological, weathering, alteration, mineralisation, structural and geotechnical information. Logged data were then entered into Excel spreadsheets using single data entry methods. Since 2012, all logging data has been captured electronically using acQuire software in the core yard, and these data are uploaded to the database upon return to the office. A stand-mounted Niton XRF instrument has been used since 2007. Pressed pellets of the prepared sample pulps are analysed to provide an initial estimate of the amount of copper present in the drill core.

Coreholes were logged at the core shed located in Kolwezi until 2009; following this all logging was moved to the Kamoa drill camp.

All drill core is photographed both dry and wet prior to sampling. All Kamoa core was subject to magnetic susceptibility measurements; these are not currently being done on Kakula core.

At Kamoa, one sample from each core run was subjected to specific gravity (SG), spectral gamma and point load testing. For Kakula, each sample length is subjected to SG testing in its entirety to ensure that every assay value has a matching SG value.

10.3 Core Handling

Core handling logs were completed that included documentation of all personnel involved in any step during the logging and sampling procedures. Transport of core boxes to the core shed was undertaken daily by Ivanhoe personnel under geological supervision. The only change to the handling of core that has been implemented since 2013 is the development of a register to track core leaving the Kakula area and arriving at the Kamoa core yard.

10.4 Recovery

Core recovery in the mineralised units at Kamoa and Kakula ranges from 0% to 100% and averages 95% at Kamoa. Where 0% recovery has been recorded at Kamoa, this is likely due to missing data, as logging does not indicate poor recovery. Visual inspection by Amec Foster Wheeler documented the Kamoa core recovery to be excellent.

Core recovery data at Kakula are generally very good, averaging 89% within the mineralised zone.

10.5 Collar Surveys

All drill sites are initially surveyed using a hand-held global positioning system (GPS) instrument that is typically accurate to within about 7 m. Prior to finalisation of a resource database, all outstanding collar surveys for completed holes that are to be included in the estimate are surveyed by an independent professional surveyor, SD Geomatique or E.M.K. Construction SARL, using a differential GPS which is accurate to within 20 mm. As of 26 January 2018, there were no outstanding collar surveys east of the West Scarp Fault being used in the Kakula Mineral Resource estimate. As of 1 November 2018, there was one outstanding collar survey (DKMC_DD1386) west of the West Scarp Fault being used in the Kakula West Mineral Resource estimate. All collars for holes used in the Kamoa Mineral Resource estimate were surveyed.

10.6 Downhole Surveys

10.6.1 Kamoa

Corehole orientations ranged from azimuths of 0° to 360°, with downhole inclinations that ranged from -5.0° to vertical. Most holes were vertical or subvertical, with only the geotechnical drillholes (-45°) and cover drillholes (<-10°) at the Kansoko Sud and Kakula declines being shallow. Downhole surveys for most drillholes were performed by the drilling contractor at approximately 30 m intervals for 2009 drilling and at a maximum interval 50 m for 2010 through 2015 drillholes, using a Single Shot digital downhole instrument. Once the hole is completed, a Reflex Multi Shot survey instrument is used to re-survey the hole to confirm the Single Shot readings.

Several coreholes were not downhole surveyed. These holes were either short holes (total depth less than 100 m) or abandoned holes, and the missing surveys do not materially impact the Mineral Resource estimate.

A total of 618 drillholes of the 776 holes used in the Kamoa resource modelling have an initial inclination of -80 to -90°, ranging in total depth from 39 m to 1,599 m. The remaining holes had initial inclinations ranging from -79° to -50°, and these holes have total depths ranging from 66.5 m to 1,271 m.

Given the relatively flat-lying nature of the mineralised units, the majority of the corehole intercepts are more or less normal to the orientation of the mineralised bodies.

10.6.2 Kakula

Downhole surveys for most drillholes were performed by the drilling contractor at approximately 3 m to 6 m intervals downhole using a Reflex Multi Shot survey instrument. In some instances, a Gyro survey instrument was used.

A total of 132 drillholes of the 155 holes used in the Kakula resource modelling have an initial inclination from -80 to -90°, ranging in total depth from 54 m to 1,448.5 m. The remaining 23 drillholes had initial inclinations ranging less than 80° to -60°, and these holes have total depths ranging from 150.5 m to 952 m. A total of 154 drillholes of the 168 holes used in the Kakula West resource modelling have an initial inclination of -80 to -90°, ranging in total depth from 233.2 m to 1,072.2 m. The remaining 14 drillholes had initial inclinations ranging from -78° to -63°, and these holes have total depths ranging from 431.7 m to 974.9 m.

10.7 Geotechnical Drilling

Where core is sufficiently competent to allow orientation surveys to be performed, Ivanhoe collects structural information for geotechnical and geological studies.

As of 1 March 2019, the database contained 50 drillholes drilled exclusively for geotechnical purposes (6,017.4 m) and 98 civil geotechnical drillholes (2,773.6 m). These geotechnical holes can form part of the resource infill grid, or are drilled as separate wedges. When drilled as part of the resource infill grid, each one had a HQ wedge of the mineralised zone that has been sampled and assayed and included in the resource estimation; the parent intersection was specifically used for rock mass characterisation testwork. The holes were drilled at various azimuths at dips between --44° and -90°.

Civil geotechnical drillholes have been used for mapping out the depth of weathering and for aiding with design of the Kakula box-cut.

10.8 Metallurgical Drilling

The location and purpose of metallurgical drillholes at Kamoa and Kakula are detailed in Section 13.

10.9 Drilling Since the Mineral Resource Database Close-off Date

10.9.1 Kamoa

The database contains 110 drillholes that post-date the Kamoa resource estimate database close-off date of 23 November 2015. Four of these holes were drilled in 2015, and 26 were drilled in 2017, with the remaining 80 holes drilled in 2018 and the first two months of 2019 assays are now available for 75 of these drillholes, and these were not used in the 2015 Mineral Resource estimate. All of these holes were drilled for resource purposes, either as infill drillholes, or resource expansion drillholes.

Of the 110 drillholes completed after the Kamoa database close-off date, two were drilled at Kansoko Sud, three were drilled at Kansoko Nord, nine were drilled south of the Makalu Dome, and 96 were drilled in the Kamoa Ouest/Kamoa North area.

Thick, high- grade intersections have been encountered at Kamoa North in the recent drilling, with follow-up drilling testing the lateral extent of these high- grade zones. These zones are characterised by a significant portion of the mineralised zone being hosted in the KPS (Ki1.1.2) (Figure 10.3).

Figure 10.3 Recent high grade intersections at Kamoa North

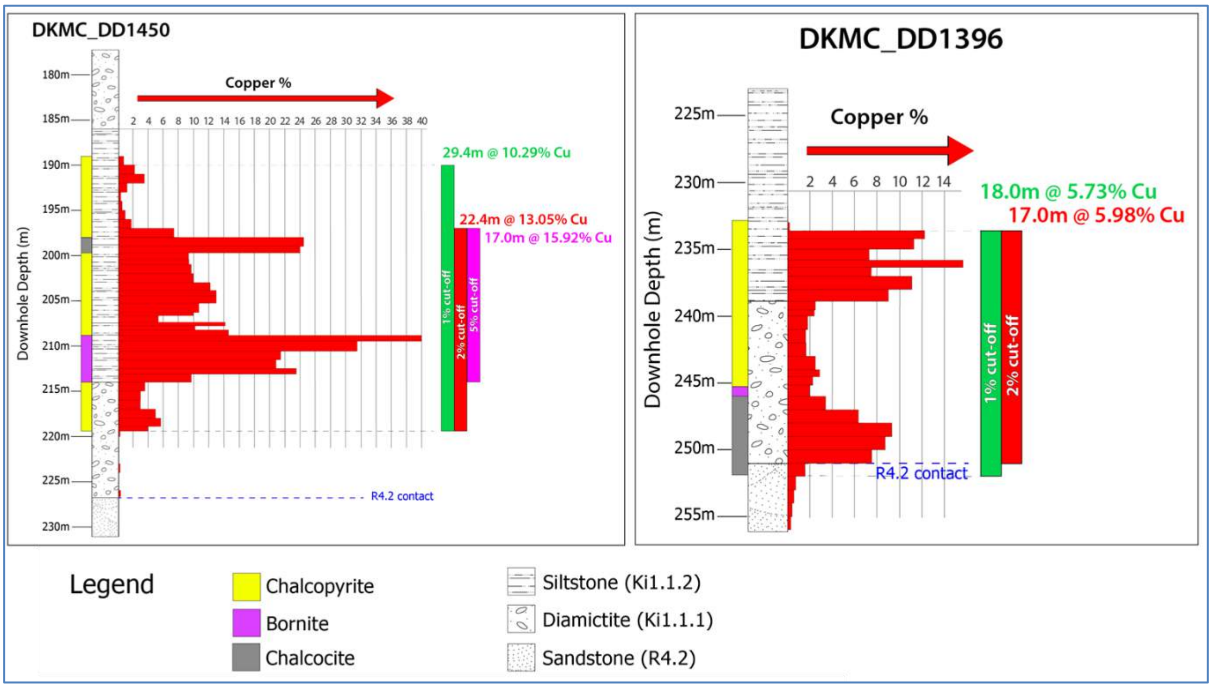


Figure provided by Ivanhoe, 2019. Red bars indicate copper grade.

Figure 10.4 shows the results of holes drilled at Kamoa Ouest/Kamoa Nord (67 holes with assay results of the total 96 holes). Figure 10.5 shows the location of the drillholes finalisation of the Kamoa and Kakula resource models.

Table 10.2 shows assay results and thicknesses for seven SMZ intercepts from the new drilling. The composite intervals shown usually do not include internal intervals of lower-grade material as is commonly found in other deposit types. The change in grade from non-mineralised to >1% Cu is usually distinct, and within the mineralised zone, grades typically remain above 1% Cu over the entire intercept. This consistency of grade is typical of the Zambian Copperbelt deposits.

Although a few of the newer drillholes are very high-grade and may change the grades locally, the majority of the holes are within the existing model and the QPs consider that the new drilling should have no material effect on the overall tonnages and average grade of the current Mineral Resource.

Figure 10.4 Plan View Showing Kamoa Drillholes with Assay Results Completed Since Construction of the 2017 Mineral Resource Model

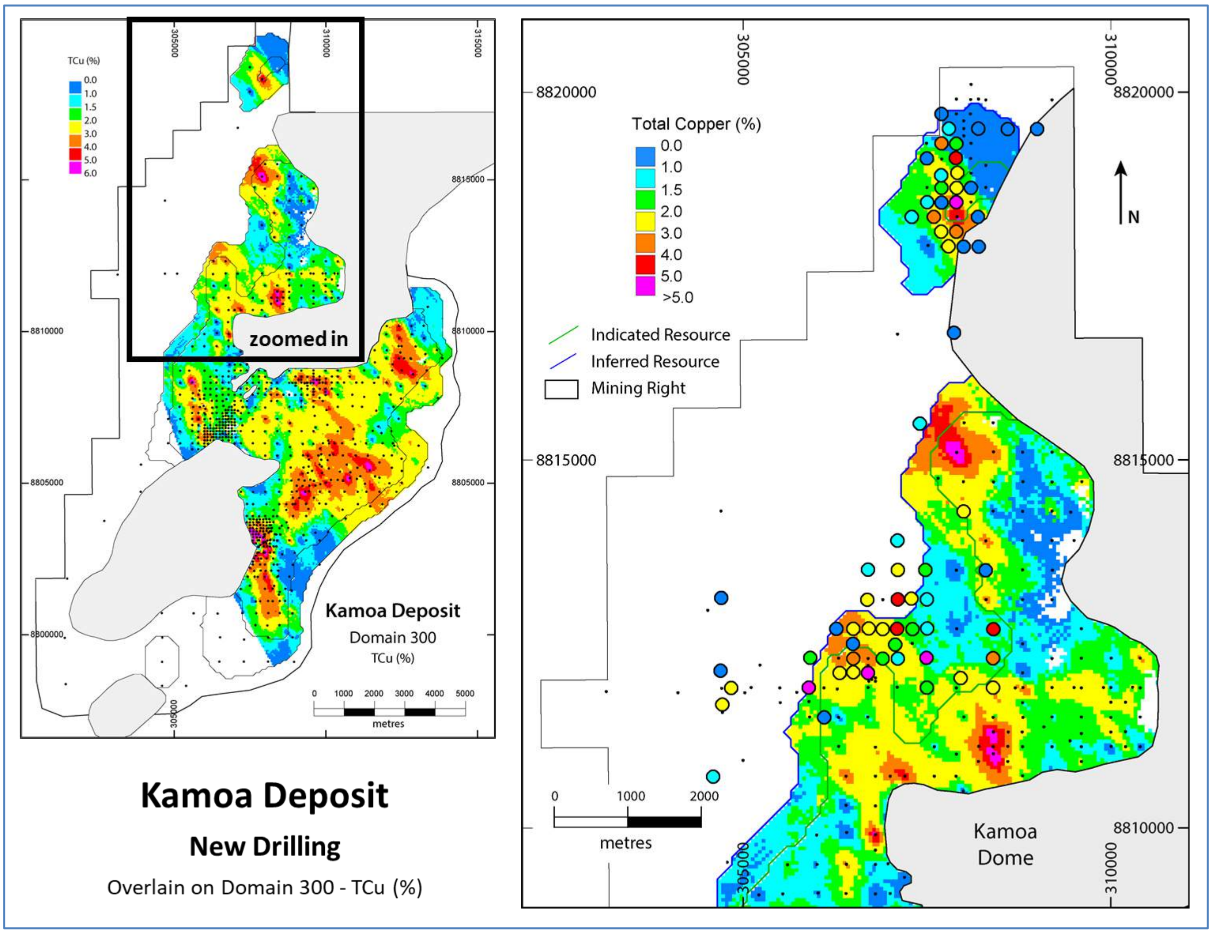


Figure by Ivanhoe, 2019. 2017 Resource model for Domain 300 is in the background; both blocks and post-resource drillholes have been colour-coded according to the legend; smaller black circles represent previously drilled holes used in the resource model.

Figure 10.5 Plan View Showing Kamoa-Kakula Drillholes Completed Since Construction of the Respective Mineral Resource Models (at 1 March 2019)

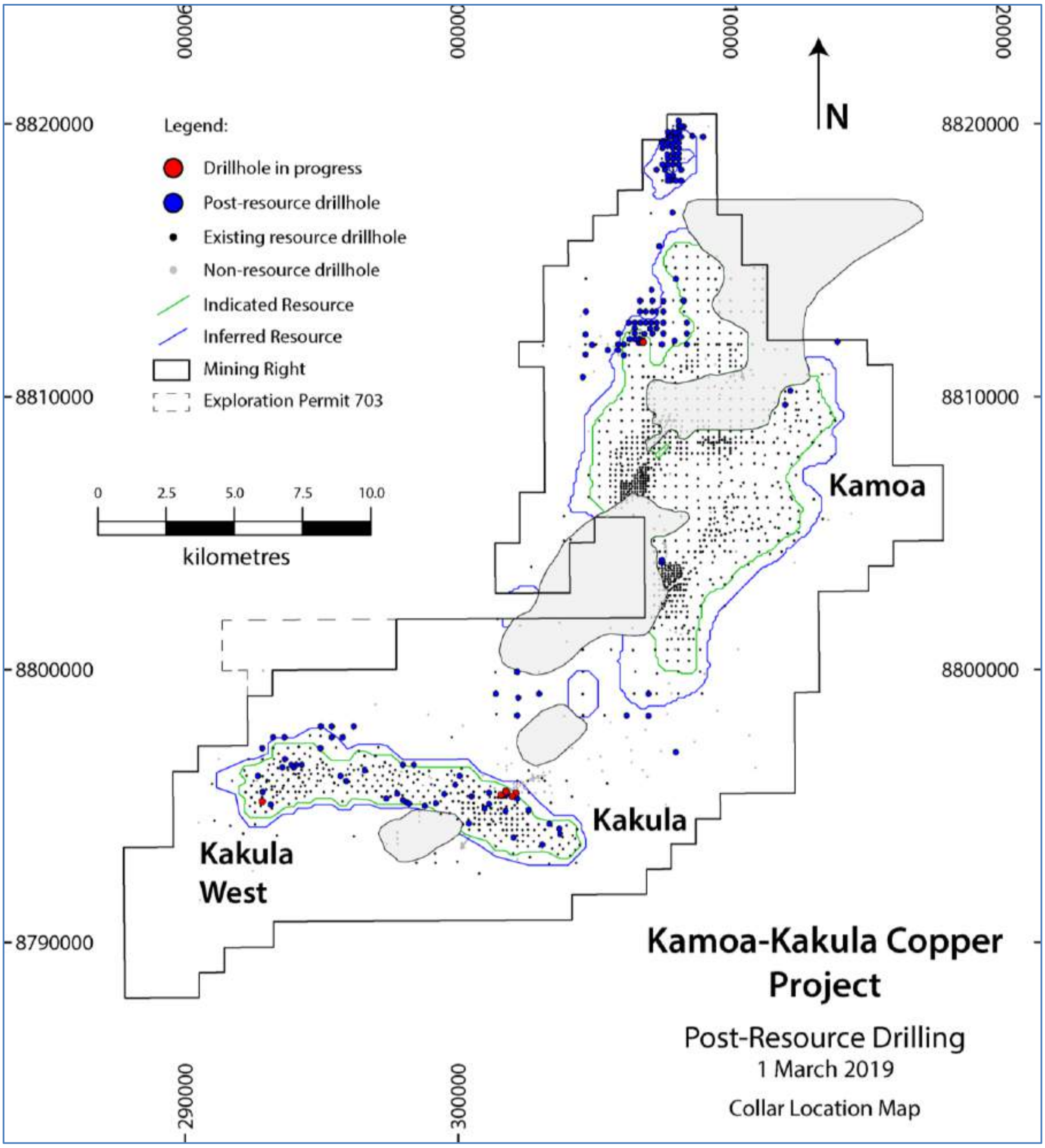


Figure provided by Ivanhoe, 2019.

Table 10.2 Example Kamoa Drill Intercept Table, Holes Drilled Since November 2014 (current as at 27 November 2017)

Drillhole ID	Easting (X)	Northing (Y)	Elevation (Z)	Azimuth (°)	Dip (°)	Total Hole Depth (m)	Intercept Depth from (m)	Intercept Depth to (m)	Drilled Intersection Length (m)	Approximate True Thickness (m)	Grade TCu (%)
DKMC_DD1183	304691.7	8812141.9	679.2	185.5	-76.5	695.7	678.00	681.30	3.30	3.20	0.63
DKMC_DD1213	307096.3	8812705.1	1167.9	351.0	-88.3	242.4	226.45	233.70	7.25	7.23	4.08
DKMC_DD1215	306709.8	8812709.8	1201.3	3.6	-89.1	203.5	178.00	181.42	3.42	3.41	2.54
DKMC_DD1222	308402.2	8812301.0	1286.1	86.7	-89.6	167.5	154.00	159.82	5.82	5.77	3.39
DKMC_DD1229	307963.1	8812032.3	1226.2	348.2	-88.5	233.6	206.00	209.60	3.60	3.58	2.13
DKMC_DD1235	306696.7	8813096.1	1230.0	325.4	-89.4	170.5	151.00	155.00	4.00	3.94	2.68

10.9.2 Kakula

Between 26 January 2018 and 1 March 2019, Ivanhoe completed an additional 20 core drillholes (11,426.2 m) in the Kakula area east of the West Scarp Fault. Assays have been received for 17 of these new drillholes. Between 1 November 2018 and 1 March 2019, Ivanhoe completed an additional 28 core drillholes (20,115.2 m) in the Kakula West area west of the West Scarp Fault. The collar locations of the coreholes are shown in Figure 10.6. The core drillholes were drilled for exploration and infill purposes. Assays have been received for 16 of these new drillholes. Drillholes completed after the database closed for the Kakula and Kakula West model, and for which assays have now been received and composite at a 3% TCu cut-off, are shown in Figure 10 6 overlain on the 3% reporting model.

Figure 10.6 Core Drilling Completed, and Assays Received, at Kakula after 26 January 2018 and at Kakula West after 1 November 2018 (as at 1 March 2019)

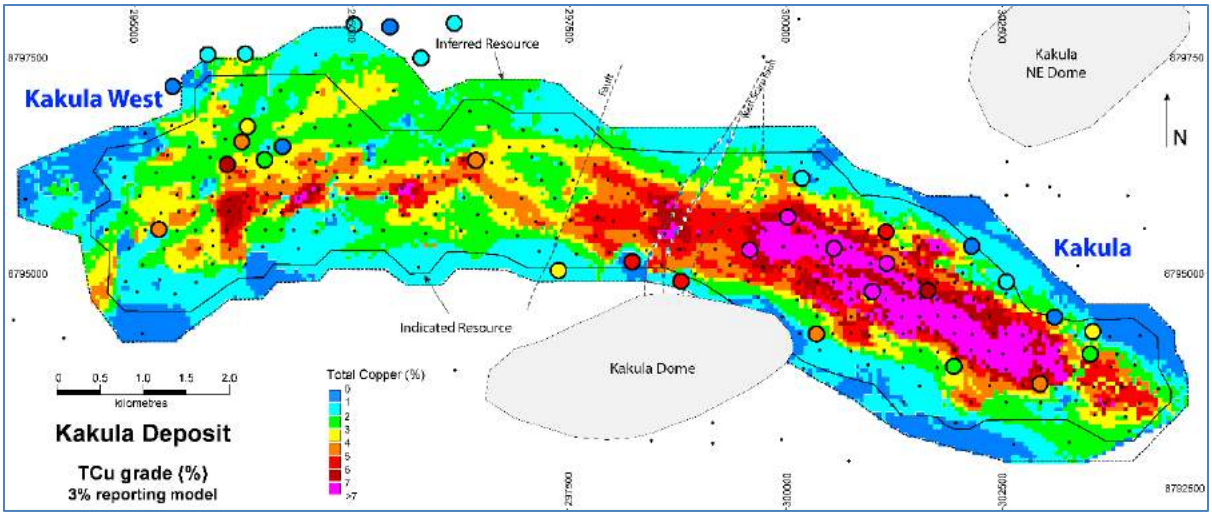


Figure provided by Ivanhoe, 2019. Both blocks and post-resource drillholes have been colour-coded according to the legend; smaller black circles represent previously drilled holes used in the resource model.

New holes within the existing Indicated Mineral Resource are infill holes that generally show similar grades as the resource model, and the QPs consider that this new drilling should have no material effect on the overall tonnages and average grade of the Indicated Mineral Resource. The seven new holes outside the the Indicated Mineral Resource model, however may slightly increase the tonnage of the Inferred Mineral Resource have upside potential for Mineral Resource estimation when incorporated into an updated model.

10.10 Comments on Section 10

In the opinion of the Amec Foster Wheeler geology and resource estimation QPs, the quantity and quality of the lithological, geotechnical, collar, and downhole survey data collected in the core drill programmes is sufficient to support Mineral Resource and Mineral Reserves estimation at Kamoa, and Mineral Resource estimation at Kakula. Specific comments are as follows:

- Drill intersections, due to the orientation of the drillholes, are typically slightly greater than the true thickness of the mineralisation.
- Drillhole orientations are generally appropriate for the mineralisation style.
- Core logging meets industry standards for sediment-hosted copper exploration.
- Collar surveys were performed using industry-standard instrumentation.
- Downhole surveys provide appropriate representation of the trajectories of the coreholes.
- Core recoveries are typically excellent.
- The SMZ can include both lower and higher-grade mineralisation; however, the transition in grade from non-mineralised to >1% Cu is usually distinct, and within the mineralised zone, grades typically remain above 1% Cu over the entire intercept.
- No material factors were identified with the data collection from the drill programmes that could affect Mineral Resource estimation.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Witness Sampling

Ivanhoe collects and maintains "witness samples", which are reference pulp samples required by the Government of the DRC for all samples being sent out of the DRC for analysis.

11.2 Sampling Methods

11.3 Geochemical Sampling

During early-stage exploration programs, the following samples were collected and used to vector into mineralisation:

- Stream-sediment samples were collected, dried and sieved. Sub-samples were submitted for analysis.
- Soil samples were collected from the B horizon depth (30 cm to 40 cm), dried and sieved. The sieved sub-samples were submitted for analysis.
- Aircore drill samples were collected from the base of each drillhole (one per hole).

Locations of all samples were recorded with a GPS. Geochemical information has been superseded by diamond drill data.

11.3.1 RC Sampling

RC samples were taken at 1 m length intervals and riffled down into two samples of approximately 1 kg each in the field using a three-stage Jones riffle-splitter, one for reference and one for homogenisation with the next metre sample, to create a 2 m composite sample.

11.3.2 Core Sampling

The core sampling procedure is as follows:

- Sampling positions for un-oxidised core are marked (after the completion of the geotechnical logging) along projected orientation lines.
- Pre-February 2010, determination of the sample intervals took into account lithological and alteration boundaries. The entire length of core from 4 m (or one core-tray length whichever is convenient) above the first presence of mineralisation and/or the mineralised zone was sampled on nominal whole 1 m intervals to the end of the hole, generally 5 m below the Ki1.1/R4.2 contact. Most intervals with visual estimates of >0.1% Cu were sampled at 1.5 m intervals or less.

After February 2010, the sampling of the KPS (Ki1.1.2) and mineralised basal diamictite was conducted as follows:

- The mineralised zone was sampled on 1 m sample intervals (dependent on geological controls).
- The Kamoa pyritic siltstone (Ki1.1.2) was sampled every 1 m, and composites were made over 3 m for analytical purposes. There is a 3 m shoulder left above the first visible sign of copper mineralisation in each drillhole.
- After March 2011, 9 m composite samples were collected in the hangingwall, and the prepared pulp was analysed by Niton. The results are used to characterise the geochemistry of the hangingwall material.
- After August 2014, whole core is logged by the geologist on major lithological intervals, until they arrive at mineralised material or at a "Zone of interest" (ZI) such as a lithology that is conventionally sampled (e.g. the Kamoa Pyritic Siltstone). The 'Zone of interest' is logged on sampling intervals, typically 1 m intervals (dependent on geological controls). Within any zone of interest, the geologist highlights material that is either mineralised or material expected to be mineralised . This is highlighted as "zone of assay" (ZA) and is extended to 3 m above and below the first sign of visible mineralisation.
- For Kakula, the KPS is not routinely sampled, as it occurs >100 m above the mineralised zone.
- Sample numbers, core quality, and "from" and "to" depths were recorded electronically on logging laptops and loaded directly into the acQuire database.
- Start and end of each sample was marked off.
- Core is cut in half for sampling (along the projected orientation lines) using an automated core cutter with diamond saw. For core likely to splinter during cutting, Pothier saw blades (thinner blades lacking any grooves) and core cradles were used. The cut line (for splitting) is typically offset from the core orientation line by 1 cm clockwise looking downhole, with the half section that contains the core orientation line retained in the core trays for geological logging and record purposes. The half-core along the right-hand side of the projected orientation lines is sampled and sent to the preparation laboratory.
- Oxide-zone samples are split using a palette knife, and the same sample protocol that is used for un-oxidised core is then applied.
- Where core is broken and cannot be cut, samplers use judgment and experience to collect half of the core from the tray. Core samplers have been trained by geologists. If large portions of the mineralised zone are broken, a wedge was drilled. If both the wedge and the parent hole had broken core, one of the intersections would be sampled in its entirety.
- One-half core samples not sent for preparation are placed in metal trays and stored at the Kamoa core shed (official core storage facility). The core storage comprises four lockable buildings with 24-hour security personnel in place. A fifth storage facility has been construction for the Kakula drillholes.

11.4 Metallurgical Sampling

11.4.1 Kamoa

The Mintek metallurgical samples were selected from available coarse reject material obtained from the corehole assay sample preparation. This material was prepared from the sawn drill core and crushed to a nominal 2 mm using jaw crushers. A quarter split (500 g to 1,000 g) is pulverised and submitted for assay. The remaining coarse reject material was retained.

The Xstrata Process Support (XPS) metallurgical samples were half HQ core; the core was then individually crushed to -3.36 mm topsize, followed by blending and sub-sampling by spinning riffler into 2 kg replicate test charges.

Upon receipt at the testing laboratories, all metallurgical test samples were placed in refrigerated storage to inhibit oxidation.

Samples collected in 2013 for Phase 4 (Open Pit) consisted of a mixture of whole PQ and half PQ core. Comminution tests used sections of full core and half core, while metallurgical tests were done on 2 x quarter core sections.

Phase 6 variability samples were collected from across the Kansoko area and are in refrigeration awaiting testing.

11.4.2 Kakula

Three metallurgical PQ holes have been drilled at Kakula through the centre of the current resource for preliminary comminution testwork.

Drilling of additional metallurgical PQ holes has been incorporated in the defined Kakula resource area to represent early years of mining and also covering up to 15 years of production. The additional PQ holes have been wedged for flotation flow sheet verification and optimisation using Kakula material. PQ holes are used for comminution testwork, while either HQ and/or NQ wedges are used for flotation testwork programs.

11.5 Specific Gravity Determinations

SG measurements were performed using a water-immersion method by Ivanhoe personnel. Samples were conventionally weighed in air and then in water. For Kamoa, density samples comprised a portion of solid core within a sample interval, and selected at intervals greater than the sampling frequency. For Kakula, all samples selected for copper analysis (from DKMC_DD1002 onwards) are also measured for SG using the entire sample interval.

At Kamoa a total of 14,754 SG measurements were performed on samples taken from drill core. Of these measurements, there are 14,753 samples with SG values between 1.5 and 4.0.

At Kakula and Kakula West, a total of 17,697 SG measurements were performed on samples obtained from remaining half core after the other half was prepared and sent to Bureau Veritas for analysis. Of these measurements, there are 117,695 samples with SG values between 1.5 and 5.0.

11.6 Analytical and Test Laboratories

Two independent laboratories have been used for primary sample analysis; Genalysis Laboratory Services Pty. Ltd. (Genalysis; from 2007 part of the Intertek Minerals Group), and Ultra Trace Geoanalytical Laboratory (Ultra Trace, from 2008 owned and operated by the Bureau Veritas Group). Both laboratories are located in Perth, Western Australia, and both have ISO 17025 accreditation.

Genalysis performed soil and stream-sediment analysis for the Project for the period 2004 to June 2005.

Subsequent to June 2005, all analyses, including drill samples, have been performed by Ultra Trace, with Genalysis used as a check laboratory for 2009 core samples.

ALS of Vancouver, British Columbia, acted as the independent check laboratory for drill core samples from part of the 2009 programme and for 2010 through 2018 drilling. ALS is ISO: 9001:2008 registered and ISO:17025-accredited.

Table 11.1 summarises the analytical laboratories names (past and present), dates used, related project/prospect/deposit, and accreditation.

Table 11.1 Analytical Laboratories Used

Original Analytical Laboratory Name	Current Analytical Laboratory Name	Dates Used	Project	Accreditation	Independent of Ivanhoe
Genalysis Laboratory Services Pty. Ltd.	Intertek Minerals Group (2007)	2004–2005 2009	Kamoa – soil and stream-sediment Kamoa – portion of check assays	ISO 17025	Yes
Ultra Trace Geoanalytical Laboratory	Bureau Veritas Minerals (2008)	2005–present	Kamoa and Kakula – all analyses	ISO 17025	Yes
ALS	ALS	2009–present	Kamoa and Kakula – check assays	ISO: 9001:2008 and ISO17025	Yes

11.7 Sample Preparation and Analysis

A mobile sample preparation facility housed in shipping containers is based on the Kamoa site and is used for all sample preparation. The laboratory is managed by Ivanhoe personnel. All drill core samples collected prior to November 2010 were processed at a similar facility in Kolwezi; subsequently (since drillhole DKMC_DD209) they have been processed at the Kamoa-Kakula site facility. The equipment at the Kamoa-Kakula facility includes two TM Terminator Jaw crushers, two Labtech Essa LM-2 pulverisers, two riffle splitters and a rotational splitter. Sawn drill core is sampled on 1 m intervals, and then the sawn core is crushed to nominal 2 mm using jaw crushers. A quarter split (500 g to 1,000 g) is pulverised to >90% -75 µm, using the LM2 puck and bowl pulverisers. A 100 g split is sent for assay; three 50 g samples are kept as government witness samples, 30 g for Niton analysis, and approximately 80 g of pulp is retained as a reference sample. The remaining coarse reject material is retained.

About 5% (approximately one in 20) of the crushed samples have a 2 mm screen test performed, and a further 5% at the pulverisation stage are checked using a 75 µm screen test. Pulp bags of the pulverised material are then labelled and bagged for shipment by air to Western Australia. From 2010, Ivanhoe has been weighing the pulp samples and records the weight prior to shipping. Certified reference materials and blanks are included with the sample submissions.

11.8 Sample Analysis

Since June 2005, all analyses, including drill samples, have been performed by Bureau Veritas Minerals Pty Ltd (Bureau Veritas, formerly Ultra Trace Geoanalytical Laboratory), with Genalysis acting as the check laboratory from 2005 to 2009. Commencing in 2010, ALS (Vancouver) took over as the check laboratory.

11.8.1 Bureau Veritas (formerly Ultra Trace) Laboratory

Bureau Veritas acquired Ultra Trace in 2007. As the assay certificates for Kamoa were certified by Ultra Trace, Amec Foster Wheeler refers to Ultra Trace in portions of this Report related to the Kamoa deposit. Assay certificates for Kakula are certified by Bureau Veritas.

Diamond drillhole samples from 2008 to February 2009 were analysed for Cu, Zn, Co (inductively-coupled plasma optical emission spectroscopy or ICP-OES), and Pb, Zn, Mo, Au, Ag, and U (inductively-coupled plasma mass spectrometry or ICP-MS) using a 4 g subsample of the pulp using an aqua-regia digest (Ultra Trace method AR105, (ICP-OES) or AR305/AR001 (ICP-MS).

From January to July 2010, drill core samples were also analysed for Ca, Co, Cr, Cu, Fe, Mn, Ni, S, and Zn (ICP OES), and Ag, As, Au, Ba, Bi, Mo, Pb, Se, Te, and U (ICP- MS) using a 4 g subsample of the pulp using mixed acid digest (Ultra Trace method ICP102 (inductively coupled plasma atomic emission spectroscopy or ICP -AES) or ICP302/AR001 (ICP-MS).

Core drill samples from January 2010 onward were also analysed for acid-soluble copper (ASCu) using a 5% sulphuric acid leach method at room temperature for 60 minutes; only 249 of the 6,640 samples obtained in 2008 and 2009 were submitted for ASCu analysis. The sampling prior to 2010 was mainly in the Kamoa area. The ASCu data are currently not used by the metallurgists; however, if the data are required for future optimisation of recoveries, a second split from these samples may be submitted for ASCu analysis. There is a risk that the samples may oxidise over time.

Samples taken subsequent to August 2010 were subjected to different analytical procedures that were requested based on the sample stratigraphic location. Samples within the KPS (Ki1.1.2) were analysed for Cu, S (Ultra Trace method ICP102 – four-acid digestion with, ICP OES), and As (Ultra Trace method ICP302, - four-acid digestion with ICP-MS). Samples within the mineralised basal diamictite were analysed for Cu, Fe, S (Ultra Trace method ICP102), Ag, and As (Ultra Trace method ICP302).

At Kakula, Bureau Veritas analysed samples for Cu, Fe, and S (BVM method ICP102 - using four-acid digestion followed by ICP-OES) and for Ag and As (BVM method ICP302 -– four-acid digestion with ICP-MS). ASCu analysis was performed on early drillholes by a 5% sulphuric acid cold leach followed by ICP-OES. ASCu analysis has subsequently been discontinued by Ivanhoe. At Kakula, no ASCu results exist for drillholes DKMC_DD1024, DKMC_DD1025, DKMC_DD1031, and DKMC_DD1033 onward.

Early drillholes (DKMC_930, DKMC_936 and DKMC_DD942) were also analysed for Au, Co, Pb, Pt, and Zn.

11.9 Quality Assurance and Quality Control

Quality assurance and quality control (QA/QC) samples are placed using between 5% and 7% insertion rate for certified reference materials (CRM), blanks and duplicates within the zone of assay, and between 3% and 5% for the zone of Interest. There are always at least two original samples before any new QA/QC insertion.

11.9.1 Blanks

Five materials, BLANK2005, BLANK2007, BLANK2008, BLANK2009, and BLANK2010 have been used in the Kamoa QA/QC. BLANK2010 and BLANK 2014 are used at Kakula. The year designations indicate the year the material for the blank was collected. A commercial low-grade CRM (OREAS22D) is also used as a blank at Kakula.

11.9.1.1 Kamoa

BLANK2005 was produced from quartz-rich material in South Africa. BLANK2007 and BLANK2008 were produced from quartz-rich material collected from a field location in the DRC. BLANK2009 was collected in the Lualaba River, about 40 km from Kolwezi. BLANK2014 was collected from the same area as BLANK2009. The material in these bags was then crushed to -2 mm ready for use as a blank in the pulverising stage of the sample preparation.

Analysis conducted at the request of Ivanhoe's consulting geochemist, Richard Carver (Carver, 2009a) revealed this material has low concentrations of the target elements Cu and Co, but the grades were not a concern.

BLANK2010 is a coarse silica material obtained from ALS; it is inserted into the sample preparation stage prior to the crushing of samples.

One blank per 20 samples was inserted prior to the samples being pulverised. Blank samples are now placed after visually-observed higher-grade mineralisation.

11.9.1.2 Kakula

Blank2010 and BLANK2014 are used as coarse blanks for the Kakula drill programme. One blank per 20 samples was inserted prior to the samples being pulverised. A pulp blank, OREAS22D, is inserted after sample preparation as it is intended to monitor analytical laboratory contamination. Blank samples are now placed after noted higher-grade mineralisation. Due to higher-grade mineralisation at Kakula, pulp blanks are currently inserted within very high-grade zones.

11.9.2 Duplicates

A preparation duplicate was created for every 20th sample by taking a second split following the crushing stage of the sample preparation. Duplicate samples are currently placed within typical mineralisation.

11.9.3 Certified Reference Materials

Kamoa uses CRMs sourced from independent companies, Geostats and Ore Research (OREAS), both located in Australia, and African Mineral Standards (AMIS), a division of Set Point Technology, located in South Africa. To date, a total of 63 commercially available CRMs has been used at Kamoa, although there are 20 CRMs commonly used. CRMs have been inserted by Ivanhoe personnel in Kolwezi, and since November 2010 have been inserted by Ivanhoe personnel at the Project site. CRMs are inserted with a 5% insertion rate, and the CRM published value is matched to the expected mineralisation grades. CRMs are placed within mineralisation to best match the surrounding material.

Kakula uses six matrix-matched and commercial CRMs to monitor the accuracy of assay performance. Matrix-matched CRMs were created using crushed materials taken from mineralised zones, were prepared by CDN Resource Laboratories Ltd., and certified by Mr. Dale Sketchley, P. Geo. of Acuity Geosciences. Commercial CRMs were purchased from ORE Research & Exploration (OREAS), and African Minerals Standards (AMIS). The AMIS CRM was not used between May 2017 and January 2018. Certified mean and tolerance limits were derived from multi-laboratory consensus programs and are used for CRM monitoring charts.

11.10 Databases

In early 2013, Ivanhoe implemented an acQuire data management database for storage of all relevant electronic data. Ivanhoe and Acuity Geoscience Ltd (Acuity) have completed validations to ensure the data integrity has been maintained during the data transfer.

Project data previously stored in various digital files were migrated into the acQuire database. Geological logs, collar, and downhole survey data are entered at the Kamoa (site) office, and assay data are imported directly from electronic files provided by the assay laboratory.

Paper records for all assay and QA/QC data, geological logging and specific gravity information, and downhole and collar coordinate surveys are stored in fireproof cabinets in Ivanhoe's Kamoa site office. All paper records are filed by drillhole for quick location and retrieval of any information desired. In addition, sample preparation and laboratory assay protocols from the laboratories are monitored and kept on file. Digital data are regularly backed up in compliance with internal company control procedures. The backup media are securely stored off-site.

11.11 Sample Security

Sample security includes a chain-of-custody procedure that consists of filling out sample submittal forms that are sent to the laboratory with sample shipments to make certain that all samples are received by the laboratory. All diamond-drill core samples were processed by the Kolwezi facility, or the onsite Kamoa-Kakula Project facility. 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. On arrival at the sample preparation facility, samples are checked, and then sample forms are signed. Sacks are not opened until sample preparation commences. Paper records are kept for all assay and QA/QC data, geological logging and specific gravity information, and downhole and collar coordinate surveys.

Transport and security procedures from the sample site to the sample preparation facilities and thence to the laboratory are discussed in Sections 11.2 and 11.7.

Half and quarter core reference samples are stored in metal trays in a purpose-designated core storage shed. Prior to July 2010, sample rejects and pulps for core, RC, and aircore samples were catalogued and stored in the Kolwezi compound. Since July 2010, all new core samples are stored at a lockable storage facility at the Kamoa site camp. All historical core has been moved from Kolwezi to the facility at the Kamoa site camp.

11.12 Comments on Section 11

In the opinion of the Amec Foster Wheeler QPs, the sampling methods are acceptable, consistent with industry-standard practice, and adequate for Mineral Resource and Mineral Reserve estimation purposes at Kamoa, and Mineral Resource estimation at Kakula, based on the following:

- Data are collected following company-approved sampling protocols.
- Sampling has been performed in accordance with industry-standard practices.
- Sample intervals of 1 m for RC drilling, and approximately 1 m for core drilling, broken at lithological and mineralisation changes in the core, are typical of sample intervals used for Copperbelt-style mineralisation in the industry.
- Samples are taken for assay depending on location, stratigraphic position, and observation of copper mineralisation.
- Sampling is considered to be representative of the true thicknesses of mineralisation. Not all drill core is sampled; sampling depends on location in the stratigraphic sequence and logging of visible copper-bearing minerals.
- The specific gravity determination procedure is consistent with industry-standard procedures. There are sufficient specific gravity determinations to support the specific gravity values used in tonnage estimates.
- Preparation and analytical procedures are in line with industry-standard methods for Copperbelt-style copper mineralisation, and are suitable for the deposit type.
- The QA/QC programme comprising blank, CRM, and duplicate samples, meets QA/QC submission rates and industry-accepted standards.
- Sample security has relied upon the fact that the samples were always attended or locked in the on-site sample preparation facility. The chain-of-custody procedure consists of filling out sample submittal forms that are sent to the laboratory with sample shipments to make certain that all samples are received by the laboratory.
- Current sample-storage procedures and storage areas are consistent with industry standards.
- Metallurgical samples have all been sourced from core rather than RC chips, and in almost all cases, properly represent uncomminuted material. Where crushed rejects have been used for Kamoa testwork, the tests performed have been on material of appropriate particle size for the test, such as flotation and ball mill grindability.

12 DATA VERIFICATION

12.1 Amec Foster Wheeler Verifications (2009–2018)

Between 2009 and 2018, Amec Foster Wheeler conducted multiple reviews of the data available to support Mineral Resource estimation.

Reviews were conducted at the end of June 2009, at the end of July 2010 (Long, 2010, Reid, 2010b), and monthly audits were performed from September 2011 to December 2012. In 2013, audits were conducted in March (Yennamani, 2013a)), August and October (Yennamani, 2013b). An audit was conducted in March 2014 (Yennamani, 2014), and in December 2015 (Spencer, 2015). In October 2016, an audit was conducted on the Kakula drillholes (Spencer, and Reid 2016), followed by an audit in May 2017 (Spencer, 2017), January 2018 (Reid, 2018a) and December 2018 (Reid, 2018b).

Reviews included checking of collar co-ordinates, drill collar elevations and orientations, downhole and collar survey data, geological and mineralisation logging, assay and specific gravity data.

No significant errors were noted that could affect Mineral Resource estimation.

As part of the data verification above, Amec Foster Wheeler reviewed the QA/QC data or QA/QC reports to ensure the assay data were of sufficient quality to support Mineral Resource estimation. The results of these reviews are discussed in Section 12.2.

12.2 QA/QC Review

Amec Foster Wheeler conducted periodic reviews of the QA/QC data between 2009 and 2013. Since 2013, QA/QC data have been reviewed by Mr. Dale Sketchley, P. Geo. of Acuity Geoscience Ltd. with the exception of the 2014 check assays, which were reviewed by Amec Foster Wheeler.

12.2.1 Kamoa Screen Tests (2009–2013)

Screen tests to monitor crusher output before splitting and pulveriser output (pulps) were routinely conducted by both the sample preparation facility on-site and by Ultra Trace; results were reviewed by Carver (2009c).

The crusher output specification is 70% passing 2 mm (10 mesh). Only 10 results from 4,446 tests were below the specification of 70% passing 2 mm. The pulveriser output specification is 90% passing 75 µm (200 mesh). A total of 760 results from 4,212 samples were below the specification of 90% passing 75 µm. A review of the samples submitted for repulverisation shows results of over 90% passing 75 µm were achieved.

12.2.2 Kamoa Certified Reference Materials (2009–2013)

Sample submissions included packets of CRMs purchased from commercial vendors Ore Research (OREAS), African Mineral Standards (AMIS) and Geostats Pty. Ltd. The primary CRMs are from OREAS and AMIS.

In the opinion of the Amec Foster Wheeler QPs, the overall relative bias for the OREAS and AMIS CRMs is within 5%, and the assay accuracy is sufficient to support Mineral Resource estimation at Kamoa.

12.2.3 Kamoa Check Assays (2009–2014)

Check assays that were performed prior to 2010 indicated that Genalysis Cu results are three relative percent to six relative percent higher than Ultra Trace for the three samples with copper grades greater than 15% Cu. This degree of disagreement is acceptable.

Subsequent to 2010, Kamoa check assays were submitted to ALS Vancouver). The Cu check assay results agree within 5%, which is acceptable.

12.2.4 Kamoa Duplicate Assays (2009–2013)

Coarse-reject (i.e. a second split of crusher output) duplicates were included in all submissions to Ultra Trace. Precision of these results indicates that better precision could be achieved by improving the crushing and splitting steps of sample preparation. A total of 90% of the pulp duplicate pairs having Cu greater than 1,000 ppm agree within 10%. Amec Foster Wheeler finds the assay precision is acceptable for Mineral Resource estimation.

12.2.5 Blanks (2009–2013)

Amec Foster Wheeler reviewed the results for 1,882 blank samples submitted for analysis. In the opinion of Ivanhoe's consulting geochemist, Richard Carver, the blank material has low concentrations of the target elements Cu and Co (Carver, 2009a). Though the results indicate that there is likely some carry over contamination of Cu at the sample preparation facility, the amount of contamination is not sufficiently high as to materially affect project assay results, and thus Amec Foster Wheeler considers that there is no significant risk to the Mineral Resource estimate.

12.2.6 2014 Kamoa QA/QC Review

Amec Foster Wheeler conducted a high-level review of the QA/QC report supplied by Acuity Geoscience Ltd. (Acuity, 2014). Ivanhoe submitted 13 certified reference materials (CRMs), blanks, and coarse reject duplicates as part of their QA/QC programme. Screen analyses are conducted to monitor sample preparation performance. QA/QC samples are placed between 5% and 7% insertion rate for CRMs, blanks and duplicates within the zone of assay, and between 3% and 5% for the zone of Interest. There are always at least two original samples before any new QA/QC insertion.

In February 2014, the spacing between jaws in the crusher improved the percent passing 2 mm from below 80% to above 90%.

Coarse reject duplicate results indicate adequate precision.

Blank samples do not indicate any sample contamination.

CRM results do not indicate any biases greater than 5%. However, Ultra Trace seems to be consistently low for CRMs (AMIS0050 and AMIS0120) with values greater than 10% Cu.

No Kamoa QA/QC reviews have been completed since July 2014.

12.2.7 Kamoa Acid Soluble Copper Determinations

In 2009, African Mining Consultants selected approximately 431 samples for ASCu analysis at Ultra Trace; of these samples, 97 were also submitted to Genalysis for ASCu determination. The ASCu results are consistent with mineralogical observations in the drill logs, with higher average values of soluble copper where the observation indicates the presence of weathering, chalcocite of probable supergene origin, or copper oxide minerals such as malachite or cuprite. Chalcocite and other sulphides are partially leached by ASCu assay procedures. The ratio of acid soluble to total copper is low (0.15 is typical) in well mineralised samples (e.g. Cu >0.5%).

Genalysis leach results were substantially lower than the Ultra Trace results. The protocol at Genalysis used a much higher ratio of sample to acid; this will slow the reaction kinetics, and has possible wetting issues (depending upon the robustness of the agitation and the tendency of the pulp to clump).

The greater excess of acid used in the Ultra Trace protocol will dissolve more partially soluble minerals. Hence Ultra Trace assays will report a higher ASCu content than will Genalysis assays, due to the differing methods.

12.2.8 Kakula QA/QC Review

Amec Foster Wheeler conducted a high-level review of the initial memorandum provided by Acuity Geoscience Ltd. (Acuity, 2016) discussing Acuity's review of the Kakula QA/QC data available as of 9 October 2016. Acuity prepared a second memorandum (Acuity, 2017) covering the period 1 July 2014 to 20 May 2017.

In January 2018, Acuity prepared a memorandum (Acuity, 2018a) discussing check assays from August 2016 to May 2017. The check assays are discussed after the CRMs.

In February 2018, Acuity prepared a memorandum (Acuity, 2018b) on QA/QC results covering the period 21 May 2017 to 28 January 2018.

In October 2018, Acuity prepared a memorandum (Acuity, 2018c) discussing check assays from May 2017 to January 2018. The check assays are discussed after the CRMs.

In November 2018 Acuity prepared a memordandum (Acutity 2018b) on QA/QC results covering the period 29 January 2018 to 2 November 2018.

Up to 20 May 2017, Ivanhoe submitted nine commercial and six matrix-matched certified reference materials (CRMs), blanks, and coarse reject duplicates as part of their QA/QC programme. Subsequent to 20 May 2017, four commercial and six matrix-matched CRMs have been submitted. Screen analyses are conducted to monitor sample preparation performance. QA/QC samples are placed at a 5–7% insertion rate for each type of quality control sample, and additional higher-grade CRMs were inserted within zones of visually high-grade copper.

Coarse reject duplicate results indicate adequate precision.

Blank samples show indications of carry-over contamination however, the values are extremely low and do not indicate any sample contamination material to the resource estimation. Review of data subsequent to May 2017 show marked decrease in carry-over contamination.

Between May 2017 to November 2018, all but one Kakula CRM returned values well within the ± 2SD tolerance limits. This single failure affected only a small number of adjacent routine samples, and the effect of this one failure on the overall quality of data is non material.CRM results indicate biases much less than 5%. Table 12.1 summarises the Kakula CRM results; Figure 12.1 shows the performance charts for KAM01, KAM02, KAM03, KAM04, KAM05, and KAM06. The Kakula assay results (RT) are shown on the right portion (BVM) of the charts. The round robin (RR) results are shown on the left portion of the charts.

Table 12.1 Kakula CRM Results

CRMs and BLKs for monitoring of Kakula copper assays at Bureau Veritas Minerals Perth, Western Australia by Method Cu_ICPOES_ICP102_ppm (21 May 2017 to 28 January 2018)

CRM Range	CRM Name	RR Cu ppm	RR RSD%	CRM Count	BVM Cu ppm	BVM Bias%	BVM RSD%
Ext. >10%	KAM06	151979	2.80	11	150182	-1.183	1.22
	KAM05	100545	2.32	30	100260	-0.278	1.12
V. High 5–10%	OREAS97	63100	5.39	6	6300	-0.158	0.72
	KAM04	53100	1.79	50	53140	0.07	1.07
High 2–5%	KAM03	48159	1.91	52	48123	-0.075	0.83
	OREAS96	39300	2.80	14	39186	-0.291	1.28
	OREAS95	25900	2.51	24	25633	-1.03	1.10
	KAM02	23214	2.07	85	23108	-0.456	0.95
Med. 1–2%	KAM01	12750	2.07	106	12788	0.131	1.30
Barren <500 ppm	BLANK2010	~15	NA	15	12	NA	NA
	BLANK2014	~15	NA	247	10	NA	NA
	OREAS22D	9	NA	73	10	NA	NA

CRMs and BLKs for monitoring of Kakula copper assays at Bureau Veritas Minerals Perth, Western Australia by Method Cu_ICPOES_ICP102_ppm (29 January 2018 to 2 November 2018)

CRM Range	CRM Name	RR Cu ppm	RR RSD%	CRM Count	BVM Cu ppm	BVM Bias%	BVM RSD%
Ext. >10%	KAM06	151979	2.80	2	150000	-1.303	0.00
	KAM05	100545	2.32	7	99771	-0.764	0.69
V. High 5–10%	OREAS166	88200	3.06	1	8700	-1.361	0.00
	OREAS97	63100	5.39	1	63900	1.268	0.00
	KAM04	53100	1.79	22	53241	0.26	1.04
High 2–5%	KAM03	48159	1.91	34	47994	-0.342	0.99
	OREAS95	25900	2.51	4	25425	-1.834	1.13
	KAM02	23214	2.07	60	23065	-0.642	1.10
Med. 1–2%	KAM01	12750	2.07	94	12774	0.027	1.24
	BLANK2014	~15	NA	186	12	NA	NA
	OREAS22D	9	NA	17	12	NA	NA

Figure 12.1 CRM Performance Chart for Kakula Matrix Matched Reference Materials

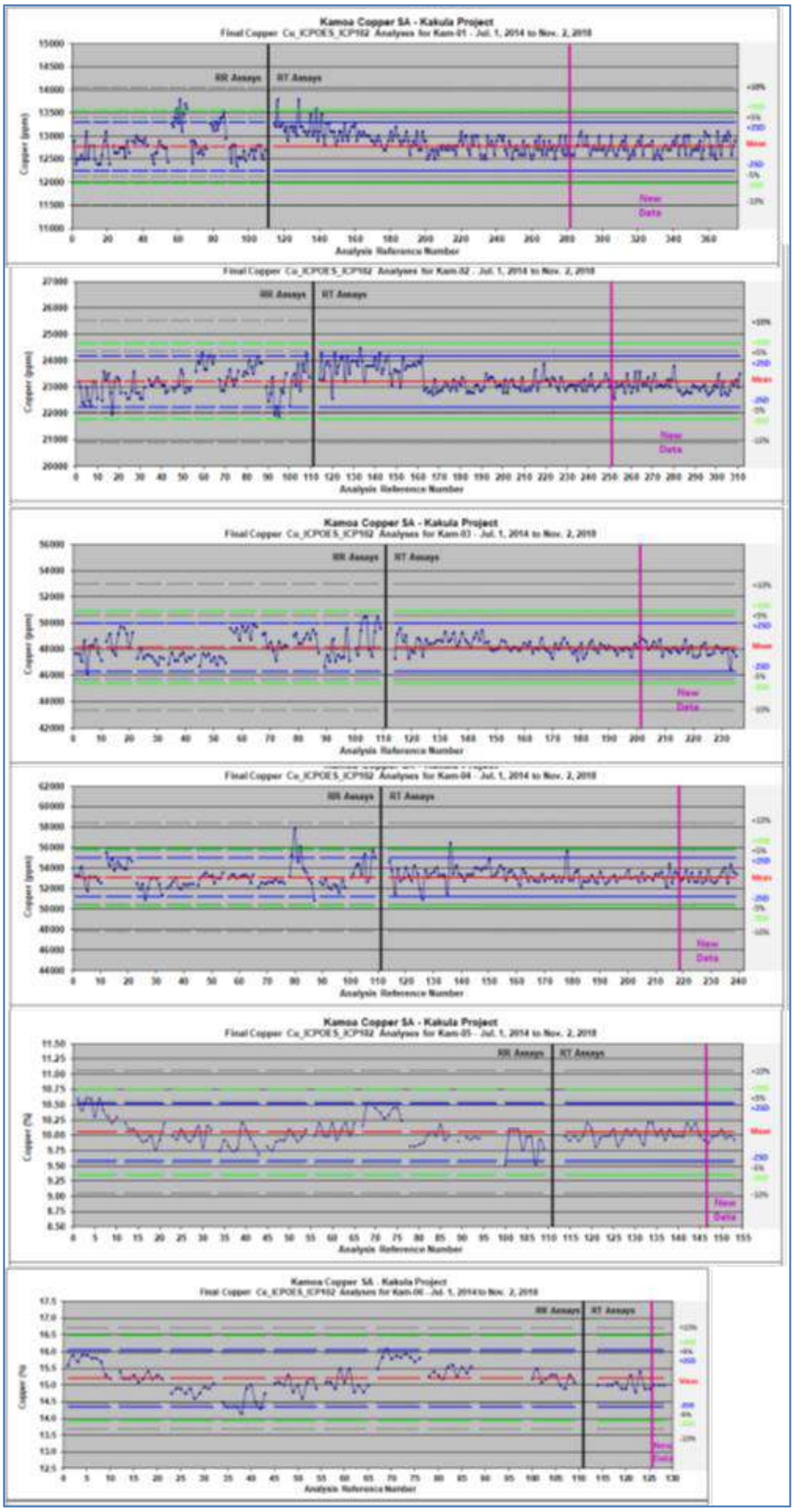


Figure by Acuity, 2018.

The check assay program consisted of reviewing data for all drilling completed at the Kamoa-Kakula Project between June 2009 and August 2016, and selecting a set of 196 representative routine samples from 50 drillholes. These samples represented five populations based on natural breaks: extreme >15%; main >6.5%; lower >2.5%; halo >1.0%; and background >0.25%. A total 20 matrix-matched CRMs, 10 blanks, and 10 pulp duplicates were inserted with an emphasis on matching grades and placing blanks after higher values. The samples were submitted to ALS Vancouver. The primary analytical method used by ALS Vancouver for the check assaying program was a four-acid digest to match that used by BVM-PT. Additionally, ALS used a sodium peroxide fusion.

A total of 277 samples was selected from 73 Kakula drillholes completed between August 2016 and May 2017 (Acuity, 2018a). These samples represented five populations based on natural breaks: extreme >15%; main >6.5%; lower >2.5%; halo >1.0%; and background >0.25%. A total of 20 matrix-matched CRMs, 15 blanks, and 10 pulp duplicates was inserted with an emphasis on matching grades and placing blanks after higher-grade values. The samples were submitted to ALS Vancouver. The primary analytical method used by ALS Vancouver for the check assaying program was a four-acid digest to match that used by BVM-PT. Additionally, ALS Vancouver used a sodium peroxide fusion.

356 samples were selected from 130 drill holes completed at Kakula between May 2017 and January 2018 (Acuity, 2018c). These samples represent the same five populations described above. A total 25 matrix-matched CRMs, 15 blanks, and 10 pulp duplicates were inserted with an emphasis on matching grades and placing blanks after higher values. The samples were submitted to ALS Vancouver. The primary analytical method used by ALS Vancouver for the check assaying program was a four-acid digest to match that used by BVM-PT. Additionally, ALS Vancouver used a sodium peroxide fusion.

The check sample assay programs conducted by ALS Vancouver laboratory validated the original BVM-PT copper assays within a normally-expected range of laboratory variations.

12.3 Site Visits

Dr. Harry Parker visited the Kamoa-Kakula Project from 1 to 3 May 2009, from 27 to 30 April 2010, from 12 to14 November 2012, and again from 17 to 19 January 2017. The site visits included presentations by Ivanhoe and African Mining Consultants' staff, inspection of core and surface outcrops, viewing drill platforms and sample cutting and logging facilities, and discussions of geology and mineralisation interpretations with Ivanhoe's staff. On his January 2017 visit, Dr. Parker checked drillhole locations, inspected drill core, and collected witness samples from the Kakula deposit. No major issues were observed. that would affect Mineral Resource estimation.

Mr Gordon Seibel visited the Project from 9 to 10 February 2011, from 5 to 8 November 2011, from 12 to 14 November 2012 and again from 18 to 22 January 2016. During the site visits, Mr. Seibel inspected drill core, reviewed drill collar locations of new drilling in the field, took independent witness core samples, inspected the on-site sample preparation facility, and observed the sampling methodology and security measures from drill stem to laboratory pickup. The site visits also included discussions of geology and mineralisation interpretations with Ivanhoe's staff, focusing on deposit strike, dip, and faulting geometries. On his January 2016 visit, Mr. Seibel checked drillhole locations at Kakula, collected witness samples, and inspected core from Kakula. No material issues were observed. that would affect Mineral Resource estimation.

12.3.1 Field Drill Collar Check

Field drill collar checks were completed by Amec Foster Wheeler staff in 2009, 2010, and 2011, 2012, 2016, and 2017 as follows:

- In 2009, Amec Foster Wheeler used a hand-held GPS unit to check the coordinates of five drillholes in the field.
- During the 2010 visit, Amec Foster Wheeler used a hand-held GPS and Brunton Compass to check the coordinates and orientation of six drillholes in the field.
- Amec Foster Wheeler used a hand-held GPS unit to check the location of 10 drillholes during the February 2011 site visit, 15 holes during the November 2011 site visit, and 11 holes during the November 2012 site visit.
- During the January 2016 visit, Amec Foster Wheeler used a hand-held GPS to check the coordinates of seven drillholes in the field.
- During the January 2017 visit, Amec Foster Wheeler used a hand-held GPS to check the coordinates of 13 drillholes in the field.

No errors were noted in the collar surveys, and all results were within the error margin of a hand-held GPS.

12.3.2 Drilling and Core Storage

Drilling was being conducted during the 2010 Amec Foster Wheeler visit, and HQ core (63 mm diameter core) was observed being recovered using an ALF-70 machine that appeared to be in good condition.

Prior to 2010, core from the barren zones was stored in aluminium boxes under tarpaulins in a field camp that was visited by Amec Foster Wheeler.

In 2010, a new core-logging facility and new secure core-storage facility were constructed at the Kamoa site. As of July 2010, all new core samples are stored at the facility.

12.3.3 Inspection of Drill Core

The following Kamoa coreholes were examined during the 2009 visit:

- DKMC series drillholes: DD005, DD006, DD007, DD008, DD014, DD015, DD019, DD023, DD034, DD040, DD041, DD043, DD046, DD047, DD052, DD053, DD056.
- DMAK series drillhole: DD004.

The following coreholes at Kamoa were examined during the 2010 visit:

- DKMC series drillholes: DD080, DD081, DD082, DD083, DD085, DD089, DD092, DD094, DD098, DD105.

The following Kamoa coreholes were examined from the KPS (Ki1.1.2) unit to end-of-hole during the February 2011 visit:

- DKMC series drillholes: DD209W1, DD213, DD215, DD216, DD219, DD221, DD223, DD228, DD229.

The following coreholes from Kamoa were examined from the KPS (Ki1.1.2) unit to end-of-hole during the November 2011 visit:

- DKMC series drillholes: DD015, DD211, DD235, DD236, DD260, DD267, DD270, DD325, DD387.

The following Kamoa coreholes were examined from the KPS (Ki1.1.2) unit to end-of-hole during the November 2012 visit:

- DKMC series drillholes: DD267, DD432, DD453, DD523, DD533, DD577, DD613.

During the January 2016 site visit, the following coreholes were examined from the Kakula deposit area:

- DKMC series drillholes: DKMC_DD924, DKMC_DD930, DKMC_936, DKMC_DD942.

Logging details were noted, in general, to match the features that Amec Foster Wheeler observed in the inspected cores.

During the January 2017 site visit, the following coreholes were examined from the Kakula deposit area:

- DKMC series drillholes: DKMC_DD997, DKMC_DD1016, DKMC_DD1026, DKMC_DD930, DKMC_DD1002, DKMC_DD1001, DKMC_DD1080, DKMC-DD1093.

Review of the eight holes showed that the identification of lithological units, alteration and sulphide mineralogy to be appropriate to provide support for resource modelling and mine planning.

12.3.4 Sample Preparation Facilities

The sample preparation facilities operated by African Mining Consultants and supervised by Richard Carver in Kolwezi were briefly examined during the 2009 and 2010 site visits.

During the 2011, 2012, 2016, and 2017 site visits, Amec Foster Wheeler toured the Kamoa-site sample preparation facility and was satisfied with the operation.

12.4 Copper Grade Check Sampling

In 2009, Amec Foster Wheeler selected 21 sample intervals from drill core boxes. The half core in the boxes was re-sawn, and quarter-core samples were taken by African Mining Consultants under Amec Foster Wheeler's direction, and submitted, along with CRMs and blanks, to SGS Lakefield.

SGS Lakefield, an independent laboratory that is located in Lakefield, Ontario, Canada, was selected by Amec Foster Wheeler to process samples, as the laboratory was not affiliated with Genalysis or Ultra Trace, and had not previously been used for sample analysis for the Project. SGS Lakefield is ISO 17025-certified, and had passed the most recent copper proficiency testing conducted by the Standards Council of Canada.

For the 2009 samples, the correlation between the laboratories was good. The ratio of the mean Ultra Trace to SGS assays for Cu was 1.01.

In 2010, Amec Foster Wheeler selected 22 sample intervals from eight drill core boxes. The half core in the boxes was re-sawn, and quarter-core samples were taken under Amec Foster Wheeler's direction, and submitted, along with CRMs and blanks, to ALS Vancouver.

ALS Vancouver was selected by Amec Foster Wheeler to process samples, as the laboratory was not affiliated with Genalysis or Ultra Trace, and had not previously been used for sample analysis for the Project.

The correlation between laboratories was found to be good. The ratios of Ultra Trace to ALS were 1.06 and 1.07 for Cu and ASCu respectively.

In February 2011, Amec Foster Wheeler selected 11 sample intervals from drill core boxes. The half core in the boxes was re-sawn, and quarter-core samples were taken under Amec Foster Wheeler's direction, and submitted, along with CRMs and blanks, to Ultra Trace in Australia. The blank and CRM (98P) results indicate acceptable performance.

Ultra Trace assayed the Amec Foster Wheeler-selected core samples for total copper and minor elements. These new results from Ultra Trace were compared to the original Ultra Trace results (ratio of witness to original assays of 0.99 for Cu without the two outlier pairs and 0.93 with the two outlier pairs).

In November 2011, Amec Foster Wheeler selected eight sample intervals from drill core boxes. The half core in the boxes was re-sawn, and quarter-core samples were taken under Amec Foster Wheeler's direction, and submitted, along with CRMs and blanks, to Ultra Trace in Australia. Amec Foster Wheeler's samples were found to be comparable to the original Ultra Trace results; Amec Foster Wheeler's Cu results were 4% lower than the original assays, while the ASCu results were 2% higher.

In November 2012, Amec Foster Wheeler selected eleven sample intervals from drill core boxes. The half core in the boxes was re-sawn, and quarter-core samples were taken under Amec Foster Wheeler's direction, and submitted, along with CRMs and blanks, to Ultra Trace in Australia. Ultra Trace's witness sample results averaged 10% lower than the original Ultra Trace assays.

In January 2016, Amec Foster Wheeler selected four sample intervals from drill core boxes from the Kakula deposit area. The half core in the boxes was re-sawn, and quarter-core samples were taken under Amec Foster Wheeler's direction, and submitted, along with CRMs and blanks, to Bureau Veritas Australia Pty Ltd. Bureau Veritas's witness sample results confirmed the presence of copper mineralisation.

In January 2017, Amec Foster Wheeler selected 20 sample intervals from drill core boxes from the Kakula deposit area. The half core in the boxes was re-sawn, and quarter-core samples were taken under Amec Foster Wheeler's direction, and submitted, along with blanks, and Amec Foster Wheeler-supplied CRMs, to Bureau Veritas in Australia. Witness-sample assay results showed good correlation with the original assays and a ratio of witness/original of 1.02 for total copper.

12.5 Comments of Section 12

In the opinion of the Amec Foster Wheeler geology and resource estimation QPs, the data verification programmes undertaken on the core data collected from the Project support the geological interpretations, and the analytical and database quality. Therefore, the collected data can support Mineral Resource and Mineral Reserve estimation at Kamoa, and Mineral Resource estimation at Kakula. Principal findings from the data verification are as follows:

- Sample data collected adequately reflect deposit dimensions, true widths of mineralisation, and the style of the deposit.
- Drill collar and downhole survey data are acceptable for use in estimation.
- The quality assurance programme for the core drilling on the Project demonstrates sufficient accuracy and precision of the copper assays for use in copper estimation.
- Matrix-matched CRMs have been created for both Kamoa and Kakula.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Testwork Overview

The Kamoa-Kakula resource has a long history of metallurgical testwork (2010 to 2015) undertaken by various parties, which focussed on the metallurgical characterisation and flow sheet development for the processing of hypogene and supergene copper ores. These investigations culminated in the development of the IFS4a flow sheet in support of the Kamoa PFS (March, 2016). During 2016, Kamoa Copper SA discovered the Kakula deposit, which has significantly higher copper head grades, compared to the Kamoa deposit. Consequently, the Kakula project was fast-tracked. Metallurgical testwork on the Kakula deposit was initiated in 2016.

For the purpose of the Kakula prefeasibility study, metallurgical testwork conducted on Kakula material has been referenced in detail. A brief summary of the historical testwork completed on the Kamoa deposit is included in this report, as background to the IFS4a flow sheet development process.

13.1.1 Historical Metallurgical Testwork on Kamoa Resource

Between 2010 and 2015 a series of metallurgical testwork programs, defined as Phases 1 to 5, were completed on Kamoa drill core sample and focussed on metallurgical characterisation and flow sheet development for processing the hypogene and supergene material. During this period the orebody was expanded, leading to major changes to mine schedules and associated processing schedules. Given that the new schedules indicated that the supergene mineralisation accounted for less than 10% of the oredoby, the focus shifted to the hypogene ores. These campaigns provided input to the development of a MF2 type flow sheet and the necessary metallurgical understanding to support the 2012 PEA and subsequent Technical Reports ahead of the Kamoa 2017 PFS (the same as in the Kamoa 2019 PFS).

In preparation for the Kamoa 2016 PFS and the increased capacity Kamoa 2017 PFS, the Phase 6 samples were selected and the associated metallurgical evaluation was conducted during 2014 2015 at Xstrata Process Support (XPS) Laboratories. The Phase 6 samples best represent ores to be processed according to the early years of the Kamoa 2019 PFS mine schedule. It is noted that many of the Phase 2 and Phase 3 samples are relevant to the current Kamoa 2019 PFS mine schedule. The Phase 6 campaign developed the IFS4a flow sheet which was confirmed as the final flow sheet for Kansoko, specifically tailored to the fine-grained nature of the material.

13.1.2 Preliminary Metallurgical Testwork on Kakula Resource

The initial mineralogical and flotation testwork on the Kakula resource was conducted during 2016-2017, at Zijin laboratories in China and XPS in Canada. Two drill core samples and three composite samples were tested, with copper head grades varying between 3.96% and 8.19%.

Mineralogical work conducted by XPS in September 2016 indicated that the main Cu sulphide mineral in the Kakula samples was chalcocite, with minor amounts of bornite and covellite. Trace amounts of chalcopyrite was detected with very low amounts of oxides. The Kakula sample was significantly higher in feldspar when compared to the Kamoa Phase 6 sample, but lower in quartz, chlorite and mica. The average grain size of the Kakula composite 1 sample was slightly coarser than the Kamoa Phase 6 sample. The Kakula composite 3 however had a finer grain size, showing variation in the Kakula material grain sizes.

The initial flotation testwork was performed by Zijin on core samples DD996 and DD998, as well as a composite sample of these cores (flotation composite sample 1). The flow sheet used for testing was a modified version of the IFS4a flow sheet referred to as IFS4b (IFS4a flow sheet with self-induced air addition). The composite sample achieved a copper recovery of 85.7% at a concentrate grade of 52.8% Cu. Following the successful testing of the flotation composite 1 sample, new samples DD1005 and DD1007 (flotation composite 2), were tested by Zijin in September, 2016; to verify metallurgical characteristics of higher-grade samples and to reconfirm if the Kakula material was compatible with the IFS4b flow sheet. A copper recovery of 85.0% at a concentrate grade of 55.6% Cu was achieved.

In September 2016, more drill cores DD1012 and DD1036 (flotation composite 3), were tested by XPS to verify metallurgical characteristics of a higher-grade sample and to reconfirm that the material was compatible with the IFS4c flow sheet (IFS4b flow sheet with adjustments of collector addition to cater for higher Cu in the sample). The flotation composite sample 3 achieved a copper recovery of 87.8% at a concentrate grade of 56.0% Cu.

13.1.3 Detailed Metallurgical Testwork on Kakula Resource

Following the successful preliminary testing of the Kakula samples, additional drill core material was tested as part of the 2017–2018 PFS testwork campaign for Kakula, which focussed on flow sheet optimisation as part of the Kakula 2019 PFS. Testwork completed during 2017-2018 included various mineralogical studies, comminution parameter testing, flotation flow sheet optimisation, HPGR testwork, concentrate and tailings thickening and filtration testing, bulk material flow testwork, comminution variability testwork, as well as preliminary flotation variability testwork.

Mineralogy studies by XPS indicated that both Kakula ore samples (2016 and 2017) tested were chalcocite rich. The PFS composite sample had higher levels of bornite and chalcopyrite compared to the 2016 flotation composite 3 sample. The main gangue minerals were quartz, feldspar, micas and chlorite. The average grain size of the Cu sulphide minerals in the Kakula PFS composite sample was finer than the Kamoa Phase 6 sample and consistent with the 2017 flotation composite 3 sample.

During 2017 and 2018, Mintek performed comminution characterisation testwork as well as preliminary variability testwork on Kakula diamictite and siltstone samples from four and six different drill cores respectively. Composite samples of the diamictite footwall and siltstone footwall were also tested. CWi values ranged from 9.8 kWh/t to 13.5 kWh/t, characterising the material as soft with regards to crushing energy requirements. The abrasion index values ranged from 0.01 g to 0.06 g with a single footwall sample measuring 0.32 g, demonstrating low abrasion tendencies of the material. The BRWi values varied between 16.1 kWh/t and 24.9 kWh/t, grouping the material in the hard to very hard classes, while the BBWi testing grouped all the samples in the very hard class with values averaging 18.2 kWh/t for the diamictite samples, and 17.5 kWh/t for the silstone samples. SMC testing also classified two samples tested as very hard, with Axb values averaging 23.0, indicating that the material was highly competent and not amenable to Semi and/or fully Autogenous Milling. The Kakula PFS samples tested had similar competency compared to the Kamoa Phase 6 material.

HPGR scoping and pilot plant testwork was conducted at ThyssenKrupp between March 2018 and October 2018 on diamictite and sandstone samples to determine key design parameters. The ATWAL abrasiveness test confirmed the low tendency to abrasiveness. The average SMALLWALL specific throughput was 285 ts/h m^3 at 3.0% feed moisture and a specific grinding force of 2.5 N/mm^2. It was noted that an increase in specific grinding force leads to a decrease in throughput – increasing the specific grinding force to 3.5 N/mm^2 resulted in a 9% decrease in throughput to 273 ts/h m^3. Higher grinding forces resulted in higher power draw – the specific energy requirement increased from 1.8 kWh/t to 2.25 kWh/t when increasing the specific grinding force from 2.5 N/mm^2 to 3.5 N/mm^2. The effect of increased moisture content was worse on the diamictite sample – an increase in moisture from 3.0% to 5.0% resulted in a throughput reduction from 28 ts/h m^3 to 267 ts/h m^3 on the diamictite sample, compared to a drop from 287 ts/h m^3 to 276 ts/h m^3 for the sandstone sample. The effect of increased moisture content did not have any impact on the fineness of the products produced. The effect of pre-screening the fines fraction from the HPGR feed resulted in lower specific throughputs – 263 ts/h m^3 for the diamictite sample and 244 ts/h m^3 for the sandstone sample. The fineness of the products produced were similar for the two samples tested.

Mintek further conducted BBWi and grindmill testing on HPGR crushed material. The BBWi, at a 75 µm closing screen, for the HPGR crushed ore, was measured at 15.8 kWh/t for the diamictite sample and 16.9 kWh/t for the sandstone sample, which was between 5% and 8% lower compared to conventionally crushed material.

Bulk material flow testing was conducted by GreenTechnical in April 2018 to facilitate with material handling designs.

XPS conducted work on the Kakula material, to further optimise the IFS4c flow sheet, following the successful results obtained during the preliminary work. Ten drill core samples were composited to form the Kakula PFS development master composite at 6.13% Cu. The scope of work included the baselining of the final grind target against the Kamoa Phase 6 IFS4c flow sheet, assessment of primary grind, and optimisation of pulp densities, reagents and reagent additions, regrind circuit, and low entrainment (dilute) cleaning. The final grind target remained at 80% passing 53 µm, as per IFS4c, however, modification was made to the air addition method from self-induced to forced air. Further, the rougher flotation feed density was increased without impacting on recoveries. Moving of the concentrate regrind step from the scavenger cleaner feed to the scavenger recleaner feed, reduced the mass reporting to the regrind circuit. A small increase in collector addition, to the scavenger recleaner stage, together with an increase in scavenger recleaner residence time, was needed to maintain recleaner recovery kinetics, as well as. Low entrainment cleaning resulted in better selectivity of copper over silica in the concentrate products. The resultant Kakula flow sheet achieved a final recovery of 85.6% Cu, while producing a concentrate product of 57.3% Cu and 12.6% SiO_2. This recovery is similar to the recovery achieved using the IFS4c flow sheet, however, an improvement in the Cu and SiO_2 grades were made.

Concentrate thickening testwork on a Kakula PFS final concentrate composite sample was conducted during July 2018, at the Outotec Testing Facility in Sudbury, to determine the optimum thickener design and operating parameters. Bench-top dynamic thickening tests indicated that an underflow solids concentration of 72.5% could be obtained from a solids flux rate of 0.25 t/m^2h. Following the thickening testwork, Outotec conducted testwork to determine the suitability of the Larox® Pressure Filter and Fast Filter Press technology for dewatering of the material. This testwork indicated that the concentrate product could be successfully dewatered to within the targeted moisture of 8%, at high solid flux rates.

Tailings settling, rheology and pressure filtration work was conducted by SGS Canada, in June 2018, to determine the optimum thickener design and operating parameters. Flocculant scoping tests indicated that the Kakula PFS sample required sequential dosing of BASF Magnafloc 380 followed by BASF Magnafloc 10. Results indicated that the tailings sample could be thickened to 59% solids w/w at a thickening area of 0.22 $m^2/(t/d)$. The rheology work characterised the sample as a Bingham plastic with a CSD of 58.5% solids (w/w) which corresponded to a yield stress of 42 Pa under un-sheared conditions, and 18 Pa under sheared conditions.

13.1.4 Kakula Preliminary Flotation Variability Testwork

Following the Kakula PFS testwork campaign, XPS conducted preliminary flotation variability testwork on the individual drill core samples from which the PFS master composite sample was constituted.

The samples tested varied from 2.6% Cu to 9.2% Cu, with sulphur grades generally increasing with increasing Cu grades. Fe, MgO, and Al_2O_3 values were relatively constant over the range of samples, averaging 5.0%, 4.0%, and 13.5% respectively. The highest arsenic value measured was 0.003% with the majority of the samples reported as below the instrument detection limit of 0.001%.

The mineralogical study indicated that the Kakula material is significantly higher in feldspar, compared to Kamoa Phase 6 sample. A varying carbonate content over the samples were noted. Chalcocite remained the main Cu minerals in all samples, with varying ratios of chalcocite, bornite, and chalcopyrite across the samples. A single sample displayed elevated levels of chalcopyrite. Sample DD1075W1 was the only sample with higher levels of poor-floating Azurite detected, and showed the lowest entitlement of sulphide Cu at 86%.

The Cu sulphide minerals that were free and liberated in the samples were low at approximately 50%. This is consistent with expectations, given the fine grained nature of the sulphides. The average Cu sulphide grain sizes varied significantly from 8 µm to 20 µm across the samples tested.

Results from the flotation testwork indicated that the chalcocite rich samples produced similar results with Cu recoveries over 80% and SiO_2 grades below 10%. The sample rich in chalcopyrite only achieved an average grade of 47% Cu product at 81% Cu recovery, and high SiO_2 at 13.8%. Sample DD1075W1 was elevated in non-sulphide Cu and achieved the lowest Cu recovery at 64.7%.

Overall, the samples tested across the Kakula deposit performed relatively consistently, on the Kakula flow sheet. The Cu mineralogy is variable and ratios between chalcocite, bornite, chalcopyrite and non-sulphide Cu are not consistent across the Kakula orebody. This variability in mineralogy resulted in changes of final concentrate grade and froth characteristics.

No correlation was noted between Cu feed grade and final Cu recovery, but did impact on the final mass pull to the product. It was observed that higher proportions of Cu was recovered in the scavenger cleaner circuit as the head grade increased. The lower feed grade samples presented poorer frothing characteristics, while the higher-grade samples benefited from longer retention times in the scavenger cleaner circuit. Given this, blending of feed material to a feed grade from 4% to 6% Cu will be beneficial for operability.

13.1.5 Preliminary Testwork on Kakula West Material

A single Kakula West sample grading 3.17% Cu was subjected to mineralogy and flotation testing at XPS in 2018. The main Cu mineral in the Kakula West material was chalcocite, followed by chalcopyrite and smaller amounts of bornite. The sample hosted higher levels of chalcopyrite than the Kakula PFS sample, with similar levels of chlorites, quartz, and mica. The Kakula West sample showed slightly lower feldspar levels when compared to the Kakula sample, but with higher carbonates. The average grain size of the Kakula West Cu sulphide minerals was noted as similar to the Kamoa Phase 6 sample - slightly coarser than the Kakula PFS sample tested.

The Kakula West sample was tested in duplicate using the Kakula flow sheet, and performed well by achieving a final Cu recovery of 86.1% while producing a concentrate at 54% Cu and 8.6% SiO_2. This indicates that the Kakula and Kakula West material can be treated in a common concentrator circuit.

13.1.6 Kamoa Phase 6 Sample Performance on Kakula Flow Sheet

In 2018 XPS tested the performance of the Kamoa Phase 6 signature plot composite sample on the Kakula 2019 PFS flow sheet to compare performance of the sample to the IFS4a flow sheet.

The Kamoa Phase 6 signature plot composite sample achieved a final Cu recovery of 86.6% while producing a concentrate at 36.2% Cu and 13.0% SiO_2. This was poorer than the sample's performance on the IFS4a flow sheet which achieved 89.3% Cu recovery while producing a product at 36.7% Cu and 9.1% SiO_2. Changes in performance can be attributed to the following variances between the Kamoa and the Kakula flow sheets:

- Better performance on the Kakula rougher/scavenger and high-grade cleaning circuit due to changes in aeration methods and additional collector (Cu losses to rougher tailings reduced from 5.6% to 4.8%).
- Inferior performance in the Kakula scavenger circuit due to repositioning of the regrind stage (increase in scavenger cleaner and scavenger recleaner tailings Cu losses from 5.0% to 8.6%).

It did however indicate that the Kakula and Kamoa material have a similar metallurgical response and that the the selected concentrator flow sheet is common concentrator to both.

13.2 Historical Testwork on Kamoa Resource

Between 2010 and 2015 a series of metallurgical testwork programs were completed on drill core samples of known Kamoa copper mineralisation. These investigations focussed on metallurgical characterisation and flow sheet development for the processing of hypogene and supergene copper mineralisation. Collectively this body of work culminated in the derivation of a MF2 style concentrator flow sheet and performance predictions (cost and concentrate production) as applied to support the PEA (2012).

During this developmental period, the known area hosting mineralisation expanded progressively, and this led to major changes to mine schedules and associated processing schedules. As an example, over time, the supergene mineralisation became less dominant and the testing focus shifted to hypogene mineralisation. Another example is that the resource and reserve grades increased as better mineralised zones were identified. Such learning and transitions are not uncommon for this style of mineralisation. The historic sample selection and testwork, defined as Phases 1 to 5, provided the requisite metallurgical understanding to support the 2012 PEA and subsequent Technical Reports ahead of the Kamoa 2017 PFS.

In preparation for the Kamoa 2016 PFS and the increased capacity for the Kamoa 2017 PFS, the Phase 6 samples were selected and the associated metallurgical evaluation was conducted over 2014–2015 at Xstrata Process Support (XPS) Laboratories. The Phase 6 samples best represent ores to be processed in the early years (Years 1 to 15) of the Kamoa PFS mine schedule, and the results will be summarised separately. Note, however, that many of the Phase 2 and Phase 3 samples are relevant to the current Kamoa PFS mine schedule.

A flow sheet was developed which was tailored to the fine-grained nature of the deposit. The circuit relied on traditional milling to P_{80} of 53 µm, followed by rougher and scavenger flotation. The concentrate streams are treated separately. The rougher concentrate was further upgraded in two cleaning stages to produce a first final concentrate stream. Scavenger concentrate, rougher cleaner and rougher re-cleaner streams were combined and ground further, to P_{80} of 10 µm in a regrind circuit. The regrind mill product was upgraded in two scavenger cleaning stages to produce a second final concentrate stream. The final concentrate stream is a combination of the rougher re-cleaner and scavenger re-cleaner concentrate streams. The final tailings stream is a combination of scavenger rougher tails, scavenger cleaner and scavenger re-cleaner tails streams. This flow sheet was confirmed as the final flow sheet for Kansoko (Kamoa) and referred to as IFS4A.

A summary of the historic testwork record prior to 2014 follows.

13.2.1 Historic Testwork Phase Definitions

The testwork programme were conducted primarily as comminution and flotation streams, and QEMScan mineralogical work was conducted to support the tests. The laboratories utilised and timing of these streams within the five historical testwork phases are shown in Table 13.1.

Table 13.1 Kamoa Historical Metallurgical Testwork

Phase	Study	Comminution	Flotation	Mineralogy	Period	Comment
1	Concept	Mintek	Mintek	SGS Johannesburg	2010–2011	Grab Samples
2	SS/PEA	Mintek	Mintek/XPS	XPS	2011–2012	Representative Composites
3	SS	–	XPS	XPS	2012–2013	Composites
4	SS	Mintek	XPS	XPS	2013	Open Pit
5	SS/PFS	Mintek	XPS/Mintek	XPS	2013–2014	Preliminary Variability

13.2.2 Historical Metallurgical Sample Locations

The drillhole locations that provided the historical Kamoa Phase 1 to 5 metallurgical samples and the PFS samples in Phase 6 are indicated previously in Figure 10.5. Many of the phase samples are localised to distinct parts of the deposit as it is now known, an indication of the evolving mine schedules. The locations of Phase 1 to 5 samples only are shown in Figure 13.1. The locations of Phase 6 samples are shown in Figure 13.5, Section 13.6 (below).

A number of the Phase 2 samples holes and a minority of the Phase 3 sample holes are in the region of the Phase 6 PFS samples. As comminution testing was carried out by area in Phase 2, some useful information for the PFS was generated at the time. No comminution testing was conducted on Phase 3 samples, which were used for flotation flow sheet development work at XPS. Three out of five Phase 5 sample holes are co-located with the area from which the Phase 6 samples were collected. Therefore, some Phase 5 results are applicable to the PFS design. Note that there were six samples tested in Phase 5 given that separate hangingwall and footwall samples were sourced and subjected to testing.

Figure 13.1 Drill Collars for Metallurgical Test Phases 1 to 5

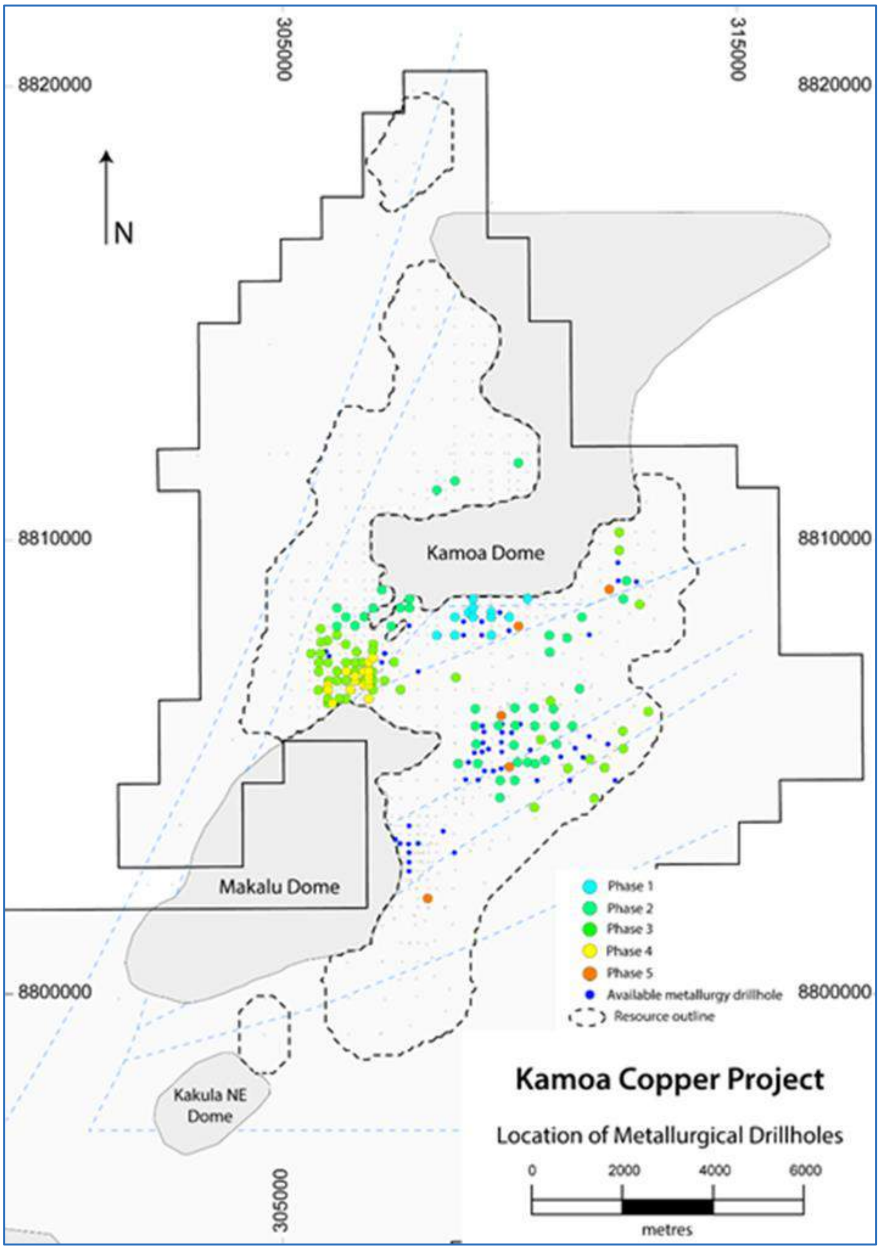


Figure provided by Ivanhoe, 2016.

13.2.3 Historical Comminution Testwork

The Phase 1 to 5 Kamoa comminution test programme is summarised in Table 13.2.

Table 13.2 Historical Comminution Programme, Sample Numbers Tested

	Bench Scale Comminution Testwork		Phase 1	Phase 2	Phase 4	Phase 5
1	SMC test		3 samples	8 samples	6 samples	6 samples
2	BRWI at 1180 µm		3 samples	6 samples	1 sample	6 samples
3	BBWI	at 212 µm	–	–	–	1 sample
		at 106 µm	3 samples	8 samples	6 samples	6 samples
		at 75 µm	3 samples	–	–	–
		at 53 µm	–	–	6 samples	6 samples
4	Ai		1 sample	8 samples	6 samples	6 samples
5	CWI		–	–	6 samples	6 samples

13.2.3.1 Competence (SMC Test) Summary

The SMC test provides measures of rock competence and grindability and is typically used for design of crushing and milling circuits, including AG/SAG milling. The range of Axb values determined on samples of various rock classes at each test phase are compared in Table 13.3.

Table 13.3 SMC Test Results as Axb Value Range

Phase	Diamictites (Hypogene, Supergene and unmineralised)	Oxide	Pyritic Siltstone (mineralised and unmineralised, hangingwall)	Sandstone (unmineralised, footwall)
1	37–38	–	29	–
2	22–31	–	21–22	25
4	–	44–58	–	–
5	17–28	–	28	30

The lower the Axb value, the harder (more competent) is the sample. Axb values below 30 indicate the sample has very high to extreme competence. Samples in the range 30 to 40 has are considered to have a high competence, whilst samples with a value above 40 have a medium competence. For reference, as no historical Kamoa samples exhibited values this high, samples with Axb values above 100 are considered incompetent.

The Phase 1 samples were taken from near-surface fresh rock and exhibited competence levels in the high range (diamictites) and at the “soft” end of the extreme range (hangingwall, typically pyritic siltstone). Samples from deeper in the deposit tested during Phase 2 were almost all in the extreme competence range. A reported value of Axb = 17 is amongst the most competent materials measured by the SMC method. The Phase 5 results, therefore, confirm the extreme competent nature of the Kamoa mineralisation (diamictites) at depth.

The samples tested in Phase 4 were selected because they represented likely open cut starter pits and represent shallow and oxidised or partially oxidised mineralised zones. All these samples fall into the medium competence range.

13.2.3.2 Fine Grindability (BBWI) Summary

The Bond Ball Mill Work Index test (BBWI) measures how difficult the sample is to grind from approximately 3 mm down to 100 µm. The index itself is a measure of the energy (in kWh/t) required to reduce the rock from infinite size to 100 µm P_{80}.

The range of BBWI values determined on samples of various rock classes at each test phase are compared in Table 13.4. Some samples exhibit different BBWI values depending on the closing screen used in the BBWI test. Where such comparative tests have been done, the results are shown separately.

Table 13.4 BBWI Test Results as kWh/t Value Range

Phase	Diamictites (Hypogene, Supergene and unmineralised)				Oxide		Pyritic Siltstone (mineralised and unmineralised, hangingwall)			Sandstone (footwall)	
Closing Screen (µm)	212	106	75	53	106	53	106	75	53	106	53
1	–	15.5	15.7	–	–	–	16.3	14.6	–	–	–
2	–	13–17	–	–	–	–	17–20	–	–	16	–
4	–	–	–	–	11–13	11.5–14.0	–	–	–	–	–
5	20	14.5–22.0	–	13.5–21.0	–	–	15.1	–	13.4	14.5	15.2

The Phase 1 and 2 samples are consistent with respect to BBWI and display slightly harder than average ball mill grindability. There is a suggestion in the Phase 2 samples that the hangingwall pyritic siltstone is harder than the diamictites. However, this is not the case with the Phase 5 samples. The footwall sandstone sample had similar grinding properties to the diamictites. The oxidised samples were consistently softer than the fresh samples.

In terms of sensitivity to grind size, fresh diamictite showed none, pyritic siltstone showed a reverse trend (i.e. softening as the grind size reduced) to that expected, and oxide showed only a slight hardening trend.

13.2.3.3 Coarse Grindability (BBWI) Summary

The Bond Rod Mill Work Index test (BRWI) measures how difficult the sample is to grind from approximately 12 mm down to 1 mm. Like the BBWI, the index itself is a measure of the energy (in kWh/t) required to reduce the rock from infinite size to 100 µm P_{80}.

The range of BRWI values determined on samples of various rock classes at each test phase are compared in Table 13.5.

Table 13.5 BRWI Test Results as kWh/t Value Range

Phase	Diamictites (Hypogene, Supergene and unmineralised)	Oxide	Pyritic Siltstone (mineralised and unmineralised, hangingwall)	Sandstone (unmineralised, footwall)
1	17–19	–	20.5	–
2	17–20	–	24.0	20.0
4	–	14	–	–
5	18–22	–	16.1	15.7

The Phase 1 and 2 diamictites are similar, as is the underlying sandstone. BRWI values in the 17 to 20 range are slightly higher than average and indicate moderate difficulty in grinding particles in a rod mill. The Pyritic siltstone result in Phase 2 of 24 kWh/t indicates a hard to very hard rod milling sample. The Phase 5 results show that some of the diamictite has very high BRWI values, and some of the bordering waste has relatively low values.

As few modern circuits contemplate rod mills, the index is most useful in providing an indication of how sensitive the ball mill will be to the presence of oversize particles in the feed. With BRWI values of 20 kWh/t the ball mill feed top size should be limited to about 9 mm. As BRWI values up to 24 kWh/t were obtained, consideration should be given to generating even finer mill feed (a topsize of 8 or even 7 mm) in the feed crushing stage.

13.2.3.4 Crushability (CWI) Summary

The Bond Crushing Work Index test (CWI) measures how difficult particles in the 50 to 75 mm range are to crush. The test does not target a product size and is complete when the particle breaks, regardless of product size distribution. Like the BBWI, the index itself is a measure of the energy (in kWh/t) required to reduce the rock from infinite size to 100 µm P_{80} using crushing. Note that although producing 100 µm P_{80} material by crushing is not practical, the definition is necessary for consistent application of the Bond comminution energy equation.

The range of CWI values determined on samples of various rock classes at each test phase are compared in Table 13.6.

Table 13.6 CWI Test Results as kWh/t Value Range

Phase	Diamictites (Hypogene, Supergene and unmineralised)	Oxide	Pyritic Siltstone (mineralised and unmineralised, hangingwall)	Sandstone (unmineralised, footwall)
1	–	–	–	–
2	–	–	–	–
4	–	8–12	–	–
5	9–20	–	16.4	9.4

The crusher work indices for shallow open pit samples are significantly lower than the deeper fresh samples, as expected. The average CWI for oxide samples was only 10.3 kWh/t while the diamictites averaged 15.9 kWh/t. It is notable that two of the four diamictite samples were above 18 kWh/t.

13.2.3.5 Abrasiveness (Ai) Summary

The Bond Abrasion Index test (Ai) measures how abrasive the sample is when it is in contact with steel. The Ai value is used to estimate consumption of steel grinding media and wear on liners of mills and crushers.

The range of Ai values determined on samples of various rock classes at each test phase are compared in Table 13.7.

Table 13.7 Ai Test Results Value Range

Phase	Diamictites (Hypogene, Supergene and unmineralised)	Oxide	Pyritic Siltstone (mineralised and unmineralised, hangingwall)	Sandstone (unmineralised, footwall)
1	0.14	–	–	–
2	0.06–0.18	–	0.04–0.05	0.38
4	–	0.01–0.05	–	–
5	0.04–0.27	–	0.15	0.08

The diamictites and the pyritic siltstone typically have Ai values less than 0.15 and all are below 0.25. These results indicate very low to low abrasiveness. The oxides also have low abrasion indices. The only sample with a high level of abrasiveness was sandstone.

13.2.3.6 Historical Comminution Characterisation Summary

The four comminution properties measured are summarised in Table 13.8.

Table 13.8 Comminution Summary by Mineralisation Type

Phase	Diamictites (Hypogene, Supergene and unmineralised)	Oxide	Pyritic Siltstone (mineralised and unmineralised, hangingwall)	Sandstone (unmineralised, footwall)
Competence	Very High to extreme	Moderate	Extreme	Very High
Crushability	Hard	Medium	Hard	Medium-Soft
Grindability – fine	Hard	Soft	Hard	Hard
Grindability – Coarse	Hard	Soft	Very Hard	Hard
Abrasiveness	Low	Low	Low	High

The high to extreme competence values means that Kamoa mineralisation is not amenable to SAG or AG milling and that crushing is the preferred coarse particle breakage mechanism. The grindability levels are suitable for conventional ball milling, and the BRWI values indicate a 7 to 9 mm Ball mill feed top size is required.

The favourable abrasiveness values in mineralised material mean the ball and liner consumptions will be low. Due care should however be taken to minimise dilution via the abrasive footwall sandstone.

13.2.4 Historical Flotation Testwork

13.2.4.1 Phase 1 (2010) – Mintek Laboratories South Africa

Mintek's Phase 1 programme was performed on drill core samples from the Kamoa Sud area of the deposit, and the tests, the first on Kamoa mineralisation, were designed to confirm amenability of the copper sulphide mineralisation to recovery by flotation. Samples were selected to represent what were the three important mineralised material types at the time. These included Hypogene, Supergene and intervals where both Supergene and Hypogene were present (Mixed). All samples were taken from a relatively shallow location close to the southern edge of the Kamoa Dome that had been extensively drilled and represented the most significant resource area in late 2009. Sample selections were made from core already drilled, logged, crushed, and sub-sampled for assay. Drillhole collar locations for the drilling used in metallurgical sampling are included in Figure 13.1.

The samples were subjected to some basic bench scale testing including grinding, rougher flotation, concentrate and tailings regrind and cleaner flotation optimisation. The separation work was supported by chemical and mineralogical analyses.

This Phase 1 flotation programme indicated:

- The mineralisation was amenable to treatment by conventional sulphide flotation, but with the provision that a significant amount of regrinding is required. Flotation recoveries were lower than typical Copperbelt ores due to a non-floating copper sulphide population locked in silicates at sulphide phase sizes of 10 µm or finer.
- The economic copper minerals identified include chalcopyrite, bornite, and chalcocite.
- Copper concentrate of greater than 25% Cu was achievable for both the Supergene and Hypogene mineralisation types tested.
- An MF2 rougher flotation scheme achieved slightly higher recoveries than a typical mill float (MF1) arrangement.
- Cleaning of concentrates after dual regrinding to 20 µm to 30 µm resulted in concentrate grades in excess of 30%, but at only modest recoveries, with the best overall result being 32% copper at 73% recovery.
- A batch testing flow sheet (Figure 13.2), which included a second stage of regrinding on middlings streams, was proposed as the go forward flow sheet concept.

Figure 13.2 MF2 Dual Regrind Circuit Flow Sheet

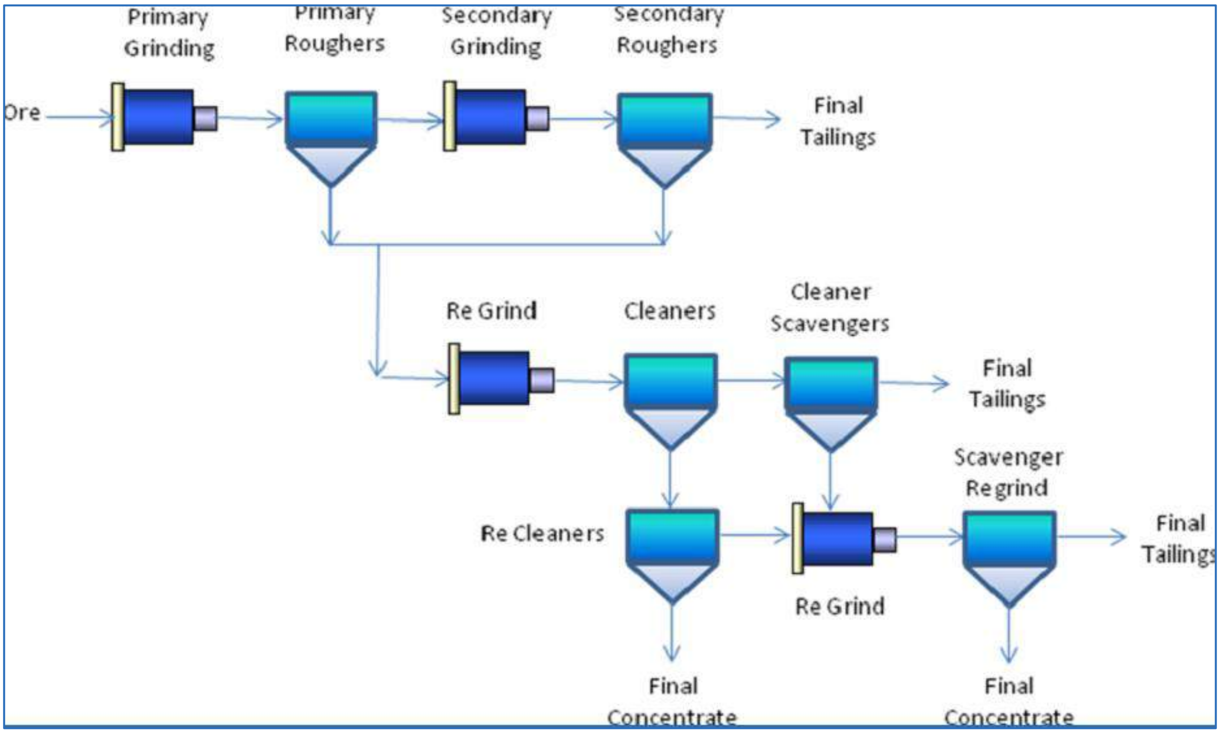


Image courtesy of Mintek 2010.

13.2.4.2 Phase 2 (2010 to 2011) - Mintek Laboratories South Africa and Xstrata Process Support (XPS) Laboratories in Canada

The resource definition drilling had advanced since the commencement of the Phase 1 work to the extent that the Kamoa mineralisation had expanded considerably by mid-2010. New samples were sourced from a range of locations with the aim of assessing comminution properties (and their natural variability) and to ascertain the robustness of the conceptual flotation flow sheet.

The flotation tests continued in development mode on composite samples and employed a relatively simple “MF2” flow sheet milling to 80% passing 75 µm, followed by rougher flotation and two stages of concentrate cleaning. The rougher tails were then reground and subjected to a scavenger flotation stage.

Phase 2 testing showed the:

- Mineralisation tested from other zones of the Kamoa deposit responded in a similar way to the Phase 1 samples, confirming that the flow sheet development direction was appropriate.
- A strong inverse relationship was found between oxide copper content and ultimate copper flotation recovery.
- The low hypogene concentrate grades confirmed that additional regrinding is necessary to achieve target.
- Copper recoveries to re-cleaner concentrate averaged only 66% for the supergene samples and 81% for the Hypogene. Concentrate grades for the supergene averaged 32% copper, but the hypogene concentrate grade was significantly lower at 17% copper.
- Although significantly different copper concentrate grades were achievable for bornite or chalcopyrite rich hypogene material (in line with sulphide stoichiometry), similar overall copper recoveries were indicated.

These Phase 2 results provided a copper grade and recovery improvement to the Phase 1 result achieved with the same Master Composite, confirming both the appropriateness of the flow sheet concept and the potential for further improvement with continued testing.

13.2.4.3 Phases 2 and 3 (2011 to 2013) – Xstrata Process Support (XPS) Laboratories in Canada

Flotation testing was moved to XPS Laboratories in Sudbury Canada during 2011.

A testwork programme was performed on drill core samples from all major areas of the expanded resource, namely, Kamoa Sud, Kansoko Sud, Kansoko Centrale and Kansoko Nord. Samples were also taken from Kamoa Ouest; however, this area did not form part of the Kamoa 2017 PFS mine plan. Composites from the Mintek Phase 2 programme were supplied to XPS to conduct comparative testing.

The composite samples were sized and subjected to mineralogical analysis using QEMScan. Parallel chemical assays were performed on the size fractions to confirm the quantitative nature of the mineralogical analysis.

Flow sheet development and optimisation testing continued during this phase. A flow sheet known as the “Milestone Flow sheet” (refer to Figure 13.3) was developed in Phase 2 that was tailored to selective recovery of the finer grained sulphide component. Similar to Mintek, the circuit relied on a mill-float-mill-float (MF2) approach to partially liberate particles, followed by fine regrinding of concentrates to achieve a concentrate grade suitable for smelting. Separate treatment of the primary and secondary rougher concentrates allowed for separately optimised cleaner flotation for coarse (fast) and fine (slow) floating minerals.

The reagent suite for the Milestone flow sheet primary consisted of a 64:36 mixture of Sodium Isobutyl Xanthate (SIBX) and dithiophosphate (Cytec 3477) added to the primary and secondary roughers, as well as the cleaners. Niche reagents Cytec 3894 and Cytec 5100 were added to the regrind mills to improve selectivity in the cleaners. Dowfroth 250 was used as the frother, and mild steel balls were used in the laboratory mills.

Figure 13.3 The Milestone Flow Sheet

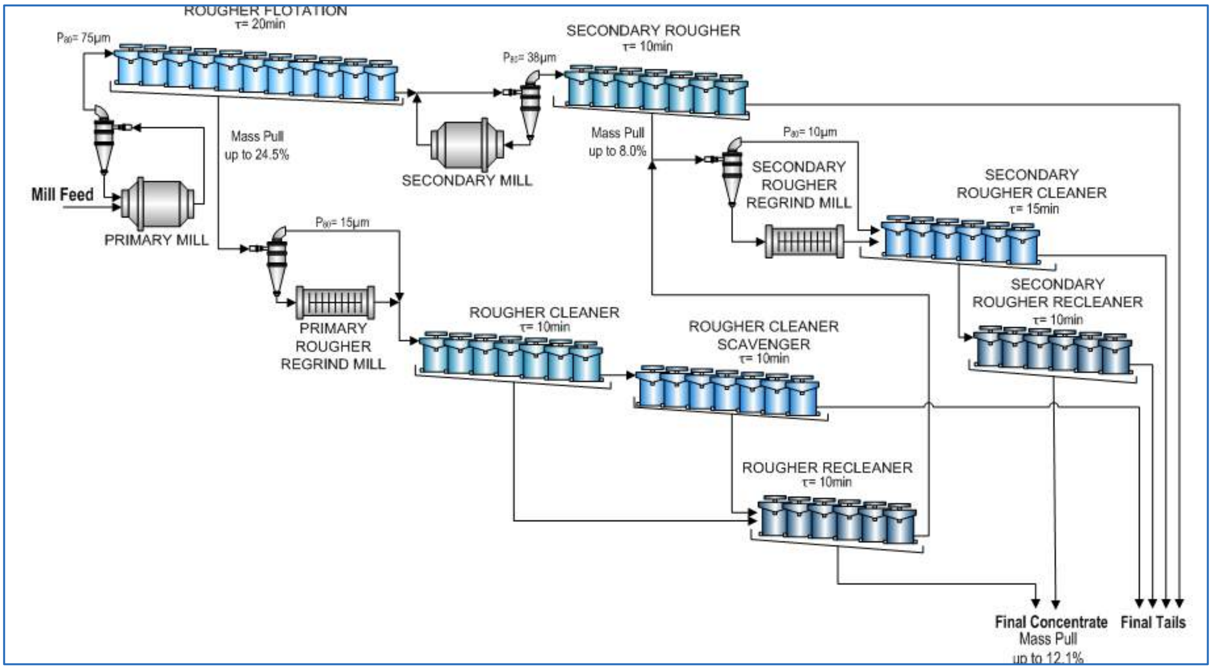


Figure by Hatch, 2013.

The Milestone Flow sheet was tested on various composites from across the resource and was able to achieve a copper recovery of 85.4% at a copper grade of 32.8% for hypogene material, and a copper recovery of 83.2% at a copper grade of 45.1% for supergene material.

In the first half of 2013 Phase 3 commenced, and the focus of development work shifted towards a reduction in the silica content of the final concentrate, in order to produce a higher quality concentrate for smelting. The ratio of SIBX to 3477 was adjusted to 85:15 to reduce silica entrainment, and the grinding media was changed to stainless steel rods in order to better simulate closed circuit ball milling with high chrome media. These changes resulted in an improvement in both the copper recovery and grade, and a reduction in silica from 19% to 13%.

The definitive flow sheet from this work stage was termed the “Frozen Flow sheet” by XPS and is shown in Figure 13.4.

Figure 13.4 XPS Frozen Flow Sheet

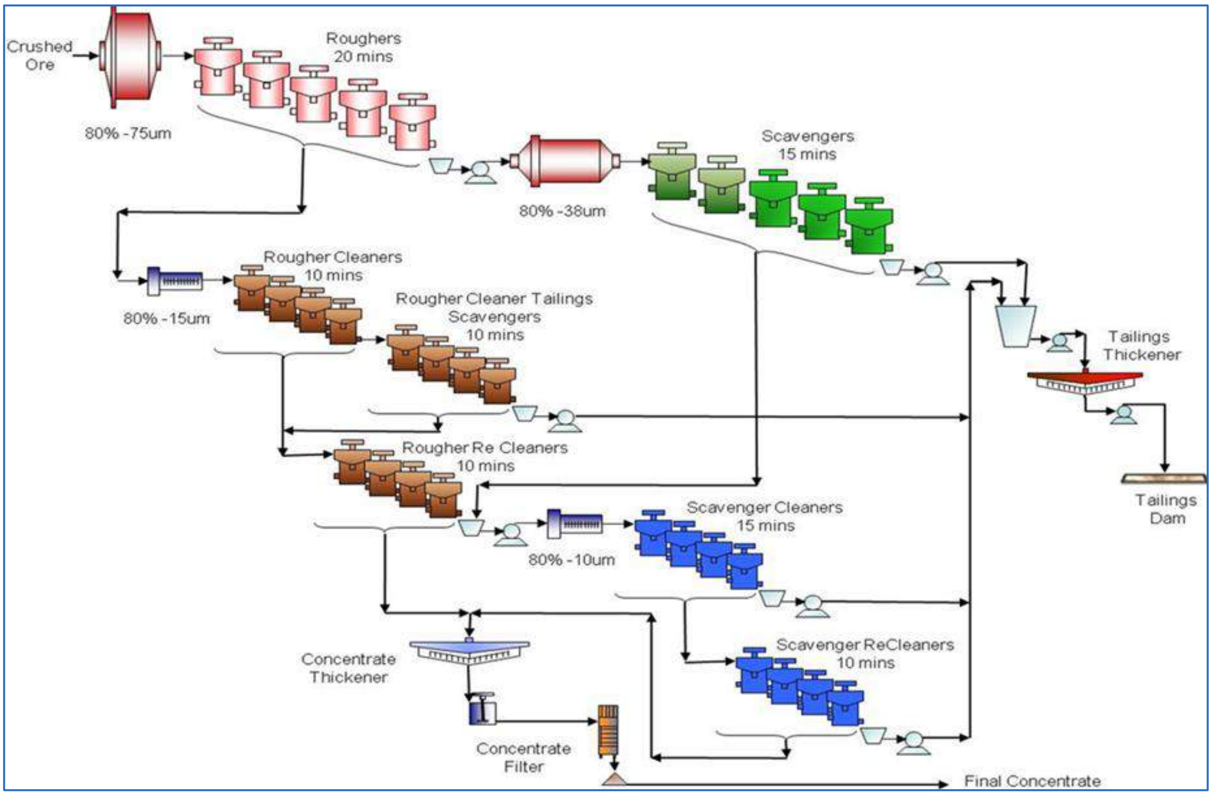


Image courtesy XPS, 2013.

This Phase 3 testwork programme indicated:

- Although significant differences were apparent in the copper mineralisation, the samples are relatively similar in terms of gangue mineralisation. The gangue minerals were dominated by orthoclase, muscovite, quartz and chlorite.
- The Supergene and Hypogene materials include a fine-grained sulphide component with more than 40% of the copper sulphide minerals having a grain size of less than 10 µm. Evidence of fine locked sulphides in silicate gangue within scavenger tails was also confirmed by QEMScan analysis.
- Chalcocite exhibits poorer liberation than chalcopyrite and bornite, which can lead to chalcocite losses in the scavenger tails and lower recoveries in the Supergene mineralisation. However, chalcocite is often found in close association with chalcopyrite rather than gangue minerals, so that 'unliberated' chalcocite can be recovered with the other copper sulphide minerals in some cases.
- Small amounts of pyrite (3.4% and 1.3% respectively) were noted in the Hypogene and Supergene composite samples. The pyrite content was determined to have been mostly contributed from samples in the Kamoa Ouest area. This pyrite content was noted to cause acidic flotation conditions which negatively affected metallurgical performance if high chrome grinding media were not used, or if a pH modifier was not added.
- In terms of copper mineralisation, the Hypogene samples tested were dominated by chalcopyrite and bornite with relatively small amounts of non-floatable azurite (<4%). In contrast, the Supergene samples tested were dominated by chalcocite and bornite and contained a larger amount of non-floatable azurite (+/-10%). This non-floatable azurite is partly responsible for the lower recoveries observed for Supergene mineralisation.
- No significant non-sulphide sulphur minerals were identified in the Supergene or Hypogene samples such that total sulphur analysis could reasonably be assumed to be equivalent to the sulphide sulphur analysis.
- Other than silica, there are no penalty elements present that reach problematic levels in the concentrate.
- Hangingwall and footwall material when mixed with the main mineralised material tended to impact concentrate quality by dilution with silica.

13.2.4.4 Phase 4 XPS Flotation Testing

The Phase 4 samples were selected from drill cores emanating from proposed open pit areas close to the Kamoa Dome and north of the Makalu Dome.

The flotation testwork showed recoveries were reasonable (80% to 87%) at concentrate grades of between 18% and 25% Cu. The main problem arising from this work was contamination of the concentrates with silica.

Open pit mill feed material does not form part of the Kamoa 2017 PFS mine schedule; thus, these results do not influence the process conclusions.

13.2.4.5 Phase 5 Mintek Flotation Testing

For a flotation method to be considered reliable it must be repeatable at a separate laboratory to the one that developed the flow sheet. Mintek was used to verify the transferability of the XPS Frozen Flow sheet and to explore some additional process options.

The XPS and Mintek performance on the same samples is compared in Table 13.9 below.

Table 13.9 Comparison of Test Procedure at Two Laboratories

Stage	Value	XPS	Mintek	Variation (%)
Feed	% Cu	4.38	4.13	-5.7
	%S	4.09	4.11	0.5
	%Fe	6.95	6.60	-5.0
Rougher	%Mass	41.7	38.7	-7.2
	% Cu	9.94	10.0	0.6
	Rec Cu	94.5	93.9	-0.6
Final Concentrate	%Mass	15.1	13.2	-12.6
	% Cu	26.3	27.6	4.9
	Rec Cu	90.8	88.2	-2.9
Tail	%Mass	84.9	86.8	2.2
	% Cu	0.47	0.56	19.1
	Rec Cu	9.16	10.59	15.6

The three excessive variations were in the concentrate mass and in the tails copper grade and distribution. The variations are magnified in the tails because of the low absolute values. The concentrate grade variation is offset by Mintek achieving a lower concentrate recovery and partially caused by Mintek's lower feed grade.

The independent laboratory repeatability testing was successful, and the method is considered transferrable and suitable for PFS design purposes, in the Frozen Flow sheet form or in later developed flow sheets having similar configurations.

Mintek conducted additional testwork but was unable to improve upon the performance achieved by the Frozen Flow sheet. Mintek made the following observations:

- An MF2 circuit at a primary grind of P_{80} 150 µm achieved higher rougher Cu recoveries as compared to the MF1 circuit at the same grind.
- The effect of grind testwork indicated that the MF1 P_{80} 150 µm cleaner test utilising coarser primary re-grind media had a potential to achieve the target specified for the Phase 5 testwork. The test had overall copper recovery of 82.9% at a Cu grade of 38.0% and SiO_2 content of 9.5%. This test indicated that copper recoveries can be further increased to obtain 85% copper recovery as the SiO_2 content was below the specified limit of less than 14%.
- The removal of the primary re-grind mill from the circuit will result in low Cu grades and high SiO_2 content in the final concentrate. This is as seen from the effect of pre-classification, single re-grinds and selective cleaning tests.
- The coarsening of the P_{80} of the primary and secondary re-grind mill products resulted in low Cu grades and high SiO_2 content in the final concentrate. This confirmed that the optimum grind for the re-grind circuit was P_{80} of 15 µm and 10 µm for primary and secondary re-grind mills respectively.
- Effect of the alternate grind test indicated that milling finer in the secondary mill increases Cu recoveries; however, this is accompanied by high SiO_2 entrainment. The secondary cleaner circuit optimisation will be required to reduce SiO_2 entrainment.

Of these observations, the most important relates to the 150 µm primary grind. A rougher flotation recovery of more than 94% was achieved by grinding to 150 µm P_{80} and floating. This compares to maximum recoveries at rougher stage of about 93%, achieved using the Frozen Flow sheet. The main penalty was additional mass recovery at the rougher stage. The rougher concentrate mass increase at 150 µm P_{80} was about 30% compared to the frozen flow sheet.

This excellent recovery at 150 µm opens the possibility for coarse primary grinding followed by staged regrinding and flotation. Mintek conducted a cleaning test based on this premise and achieved a concentrate grade of 34.9% Cu at a recovery of 84.3%. This compared with Mintek's baseline test result of 34.7% Cu at a recovery of 85.7%. Note, however, that the coarser primary grind offers little practical advantage because both circuits consume about 26.5 kWh/t of new feed when all regrinding is included.

13.2.5 Kamoa 2017 PFS Design Testwork

To support the Kamoa 2017 PFS, samples were collected from probable mining areas. These samples were subjected to comminution testing at Mintek and flotation testing at XPS.

13.2.5.1 Phase 6 Comminution Testwork – Mintek

Samples were collected for comminution testing. The samples consisted of hangingwall composites, footwall composites and variability samples from what has been termed the Minzone. Minzone refers to the single 6 to 12 m thick mineralised zone which is a consistent feature at all locations across the Kamoa deposit. Minzone samples have been prepared on the basis that the entire mineralised zone from a given location will be mined and processed together. Even if there are a variety of domain types within the Minzone at a particular location, it will not be possible to mine and process them selectively.

The samples collected specifically for PFS testing in Phase 6 were taken from holes selected on the basis of the 2013 PEA mine plan. The locations of these samples are shown in Figure 13.5 together with the early PFS mining areas. Samples from the 6A set have been used in comminution testing, and both 6A and 6B samples have been used in flotation testing. The Phase 6 comminution results are shown in Table 13.10.

Figure 13.5 Drill Collars for Phase 6A and 6B Samples

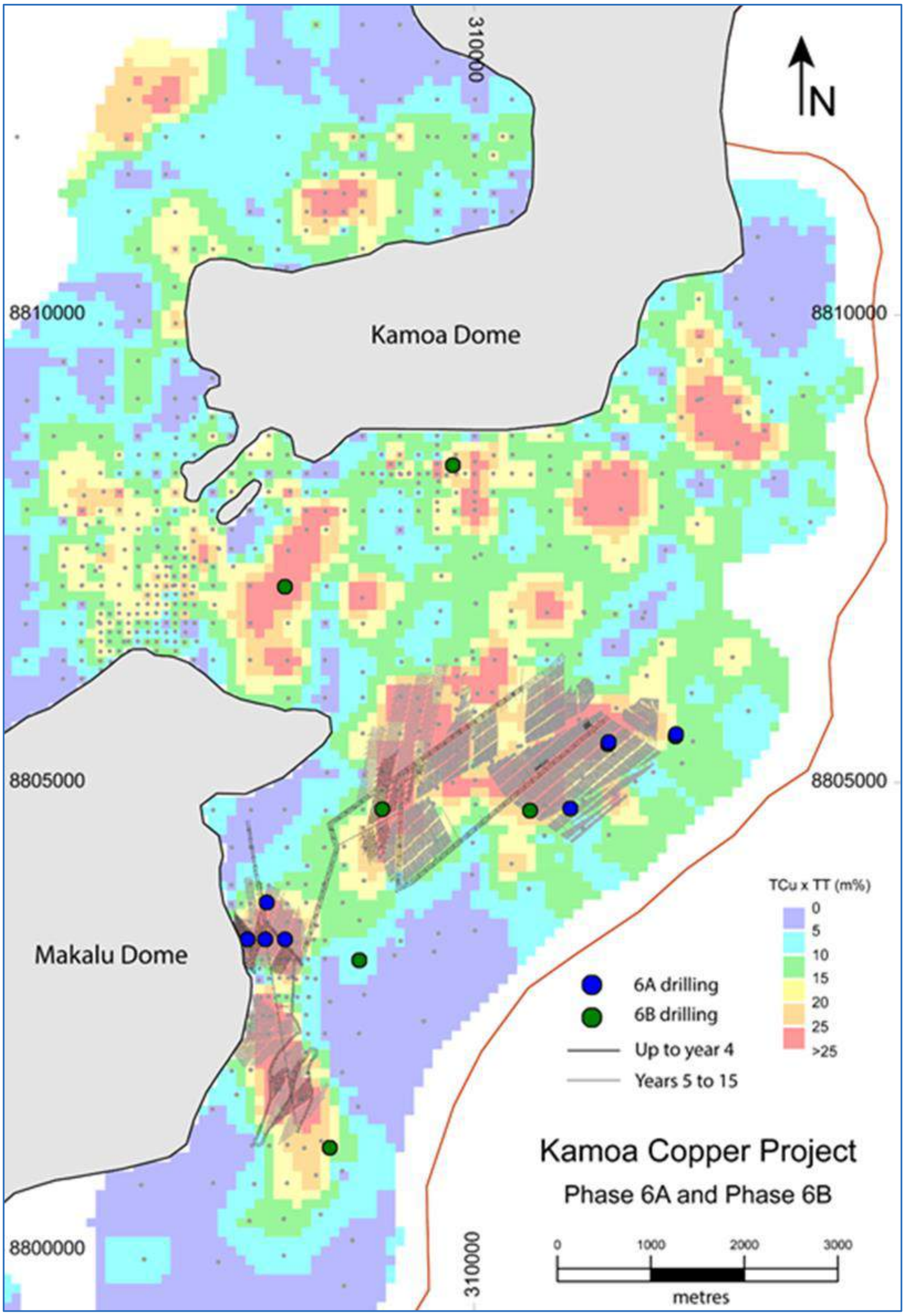


Figure provided by Ivanhoe, 2016.

Table 13.10 Phase 6 Comminution Summary

Sample ID		BRWi	BBWi (kWh/t)		UCS (Mpa)	CWI (kWh/t)	Ai	
	SG	kWh/t	53 µm	106 µm	Avg	Avg	g	A*b
HW Sandstone Composite	2.43	10.8	14.6	15.4	36	9.1	0.07	–
HW Diamictite Composite	2.82	21.1	15.9	17.3	169	9.4	0.04	–
DD345 W3 Minzone Diamictite	2.83	21.5	18.1	20.8	162	10.9	0.11	–
DD357 W7 Minzone Diamictite	2.85	23.3	19.9	19.4	140	10.7	0.07	–
DD445 W2 Minzone Diamictite	2.85	22.8	18.8	19.4	178	10.8	0.07	–
DD858 W2 Minzone Siltstone	2.58	18.4	13.3	14.2	113	7.2	0.04	–
DD859 W2 Minzone Diamictite	2.77	22.2	18.1	17.3	202	10.4	0.04	–
DD860 W2 Minzone Sandstone	2.27	11.2	11.5	12.1	39	8.5	0.03	–
DD864 W2 Minzone Diamictite	2.74	19.6	16.9	16.3	122	7.8	0.03	–
FW Diamictite Composite	2.78	20.2	16.2	16.3	129	7.8	0.08	–
FW Sandstone Composite	2.76	20.4	18.3	18.8	296	20.3	0.38	22.5

These results are compared with the historical values in Table 13.11. Note that there was one sandstone and one siltstone sample in the Minzone variability set, and that each of these was only assigned a one eighth weighting when determining average properties for their respective rock types. The hangingwall and footwall composites are each prepared from core adjacent to the seven Minzone samples and were given a weighting of seven eighths in the calculations.

Table 13.11 Comminution Properties

Mineralisation Type	Measure	Phase 6 (PFS) Average Value	Overall Historical Summary	Consistent
Diamictite	Axb	–	17 to 38	–
	BBWI (106 µm)	17.7	13 to 22	Yes
	BRWI	21.5	16 to 23	Yes
	Ai	0.060	0.04 to 0.27	Yes
	CWI	9.7	9 to 20	No
	UCS	119	95 to 255	Yes
Siltstone (Hangingwall)	Axb	–	21 to 29	Yes
	BBWI (106 µm)	15.7	16 to 20	Yes
	BRWI	11.8	20 to 24	No
	Ai	0.069	0.04 to 0.05	Yes
	CWI	8.9	16.4	No
	UCS	43	95	No
Sandstone (Footwall)	Axb	22.5	25	Yes
	BBWI (106 µm)	18.0	16	Yes
	BRWI	19.3	20	Yes
	Ai	0.334	0.380	Yes
	CWI	18.8	9.4	No
	UCS	190	–	–

There are four instances where the Phase 6 results are not consistent with the historical results. Three instances are in hanging or footwall comparisons and are based on one or two results in each instance; thus, these inconsistencies are not material for design thinking. The most important mismatch instance is in the Minzone and it is the CWI value. According to the seven Phase 6 samples the CWI is consistently in the range 7.2 to 10.9 kWh/t. In contrast the four Phase 5 Minzone samples vary from 9 to 20 kWh/t. Of more concern is that the two Phase 5 samples in the PFS mining zone (as all the Phase 6 samples are located in the PFS mining zone) have CWI values twice that of the Phase 6 samples at 18.6 and 19.6 kWh/t respectively.

The Kamoa 2017 PFS basis of design (BOD) uses the comminution properties in Table 13.12. An appropriately high CWI value has been selected.

Table 13.12 Design Comminution Properties

	BOD	Selection Method
Axb	18.1	UCL90 + SD
BBWI (kWh/t) at 53 µm	20.8	Maximum (diamictite)
BRWI (kWh/t)	23.3	Maximum (diamictite)
Ai	0.08	UCL90
CWI (kWh/t)	18.1	UCL90 + SD

The UCL90 is a statistically determined value from the available data and is explained graphically in Figure 13.6. The points on the graph are the fourteen measured values for Ai on underground samples (Phases 2, 5, and 6).

Figure 13.6 UCL90 Determination for Ai

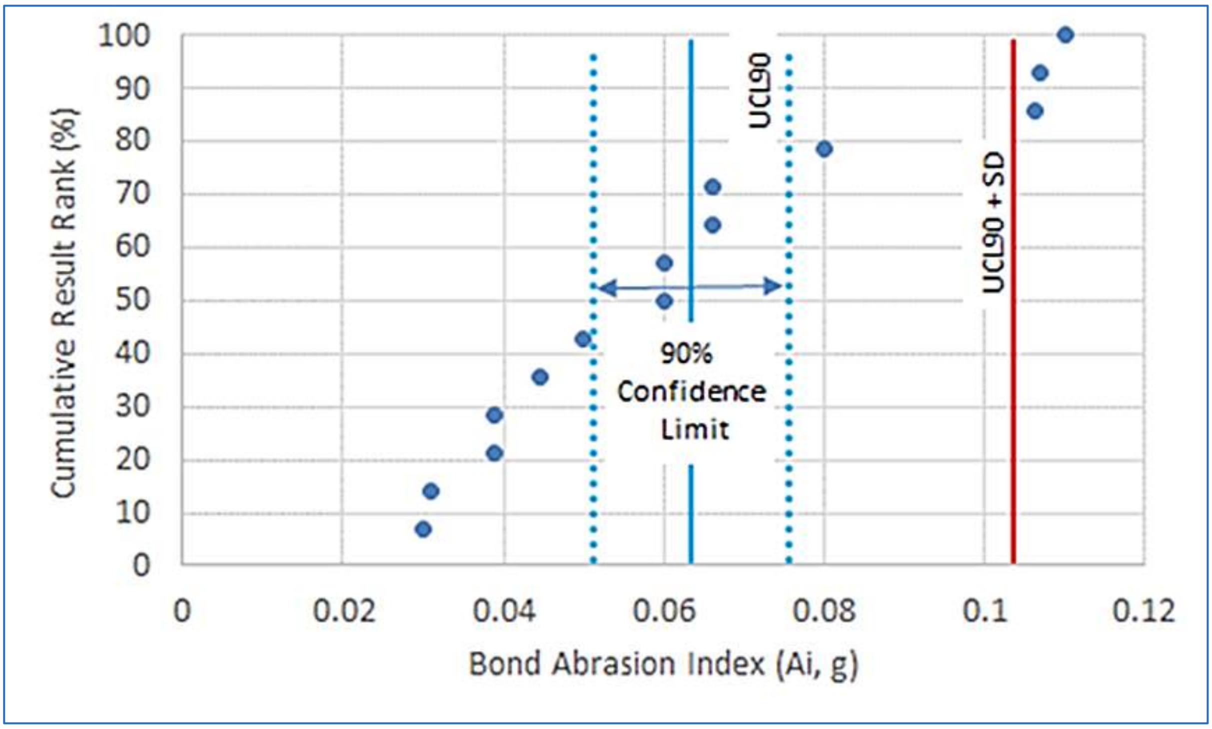


Image courtesy of Amec Foster Wheeler, 2016.

The mean value for the set is Ai = 0.063. The confidence limit is a measure of how confidently the mean or average value has been measured by the testing actually performed. As more samples are tested, the measurement of the mean value improves. Practically speaking, it means that If the same number of samples were chosen and tested again for Ai from all the available samples, then 9 times out of 10 (90% of the time) the mean result should fall within the confidence limits. Therefore, the UCL90 is a reasonable estimate for a safe mean value, where the mean is a required input for design.

13.2.5.2 Phase 6 XPS Flotation Testing

The Phase 6 XPS testwork programme was designed to establish the performance of the preferred flotation flow sheet on the ores that form the early years of Kamoa 2017 PFS mine schedule.

Composites representing Years 0 to 4 were tested under the label Phase 6A, and composites representing Years 5 to 15 were labelled Phase 6B as indicated in Figure 13.7.

Figure 13.7 Drill Collars for Phase 6 Flotation Test Composite Samples

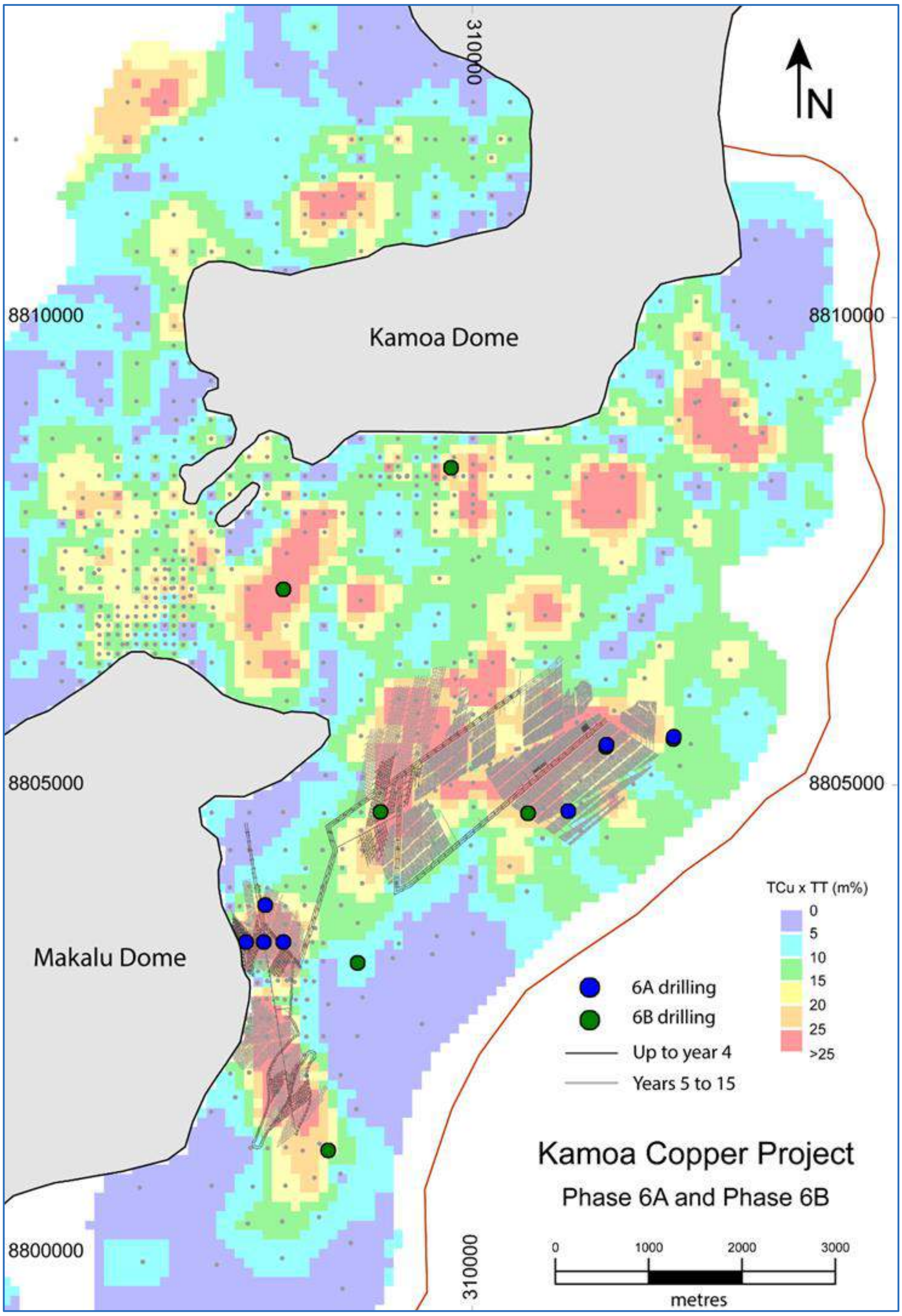


Figure provided by Ivanhoe, 2016.

The Phase 6 samples were prepared in sets containing a development composite (DC) and two individual composites based on copper sulphide mineralisation classification. The composite head assays are contained in Table 13.13.

Table 13.13 Phase 6 Flotation Test Composites

Phase	Sample	% Cu	% S	% Fe	%CaO	%Al2O3	%MgO	%SiO₂
6A	6A1 DC	3.67	2.21	5.21	0.65	12.5	2.77	63.3
	Hypogene	3.57	3.08	5.43	0.28	13.0	2.82	61.5
	Supergene	3.68	1.07	5.13	0.06	12.8	2.29	61.0
6B	6B1 DC	3.27	2.57	5.52	3.97	12.2	3.93	63.4
	Hypogene	2.99	1.70	4.64	0.71	12.6	3.51	62.7
	Supergene	3.87	1.15	4.84	0.05	11.5	1.83	66.3

One distinguishing factor between the various composites is the ratio of copper to sulphur as shown in Figure 13.8.

Figure 13.8 Copper to Sulphur Ratios in Phase 6 Composites

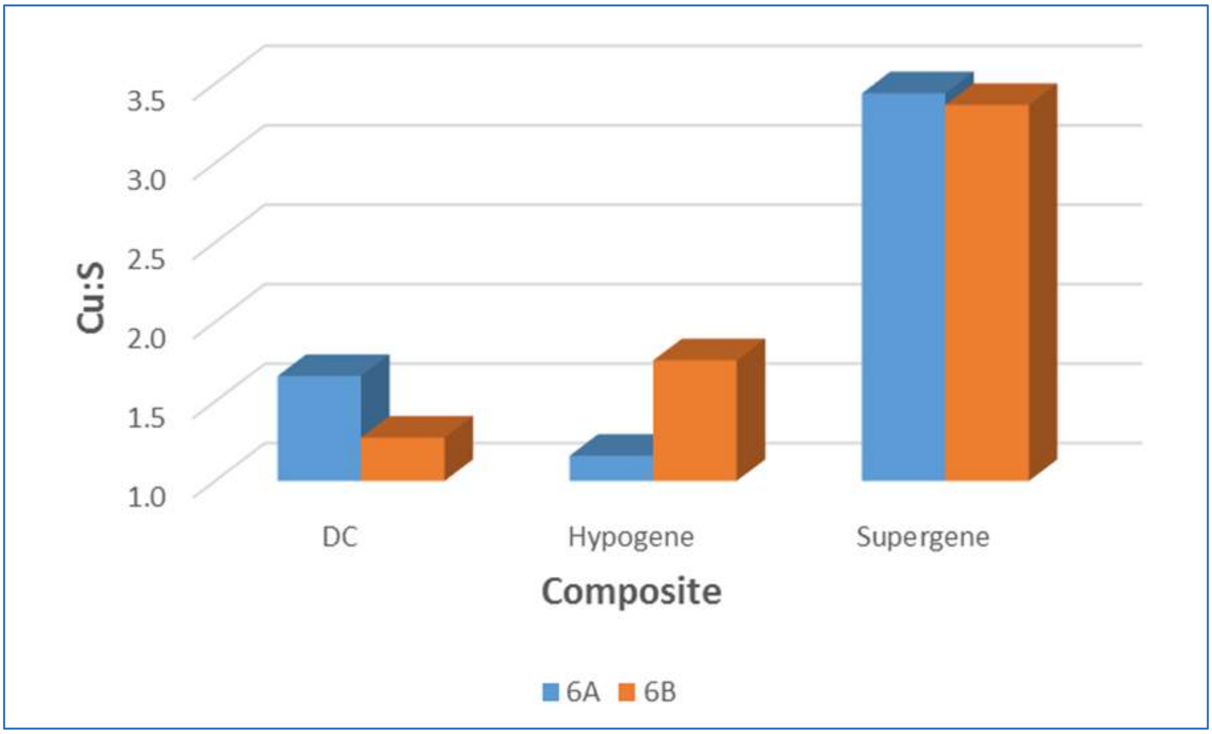


Image courtesy of Amec Foster Wheeler, 2016.

Normally, hypogene would have the lowest Cu:S ratio of the three composite types as it is usually dominated by chalcopyrite and is likely to have some pyrite present. This is the case for the 6A sample set. However, the hypogene and DC composite Cu:S ratios are opposite to expectations. In the 6B sample set the copper mineralogy of the hypogene composite is dominated by Bornite while the DC sample is dominated by chalcopyrite and pyrite.

Supergene mineralisation consists of sulphur poor copper minerals such as chalcocite and covellite as well as sulphur free minerals such as malachite and azurite. The proportions of these minerals present are clearly shown in Figure 13.9. This leads to the high Cu:S ratios shown in Figure 13.8.

The Cu:S ratio anomalies for the hypogene and supergene composites are explained by the QEMScan mineralogical analysis in Figure 13.9.

Figure 13.9 QEMScan Copper Mineralogy of Phase 6 Composites

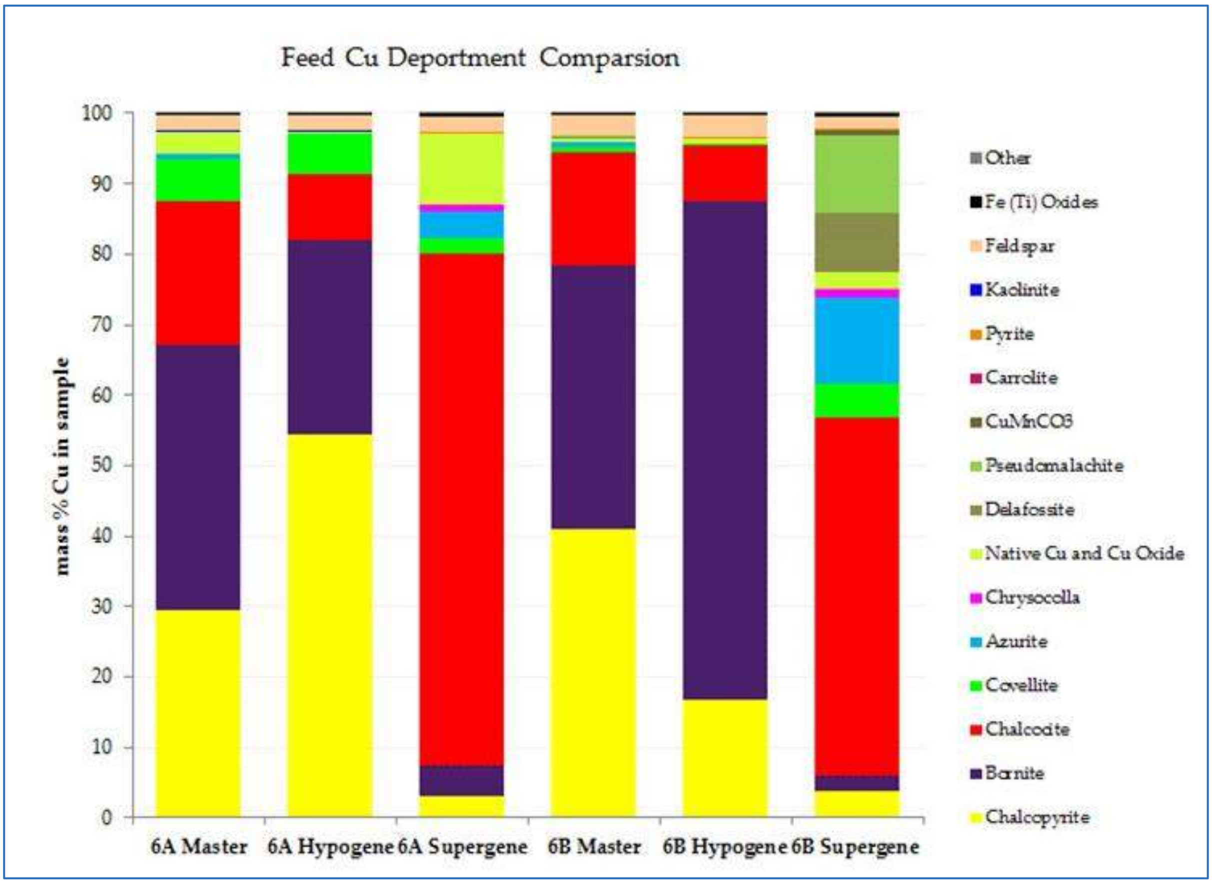


Image courtesy XPS, 2015.

Master sample is an alternative name for DC sample. The DC samples both have a mix of hypogene and supergene. The presence of supergene in the 6B Master sample is best illustrated by the presence of azurite, which is always absent in Kamoa hypogene. The purple band represents bornite which has a relatively high Cu:S ratio. It is the dominance of bornite in the 6B hypogene sample that leads to its anomalous Cu:S ratio. The final flow sheet format used to test and compare these samples is termed by XPS the "Integrated Flow sheet" or "IFS". This is an MF1 or Mill-Float style circuit (as opposed to the earlier MF2 circuits) and recovers both coarse (53 µm P_{80}) and fine (10 µm P_{80}) concentrates. The initial form of the flow sheet also has a rougher tails coarse scalping stage, a feature that did not persist into the final test flow sheet or the Kamoa 2017 PFS flow sheet. A number of versions of this flow sheet were tested, and the preferred configuration was termed IFS4. The IFS4 flow sheet is shown in Figure 13.10. Each of the six primary Phase 6 composites was tested using this flow sheet and the results are compared in Table 13.14.

Figure 13.10 XPS IFS4 Flow Sheet

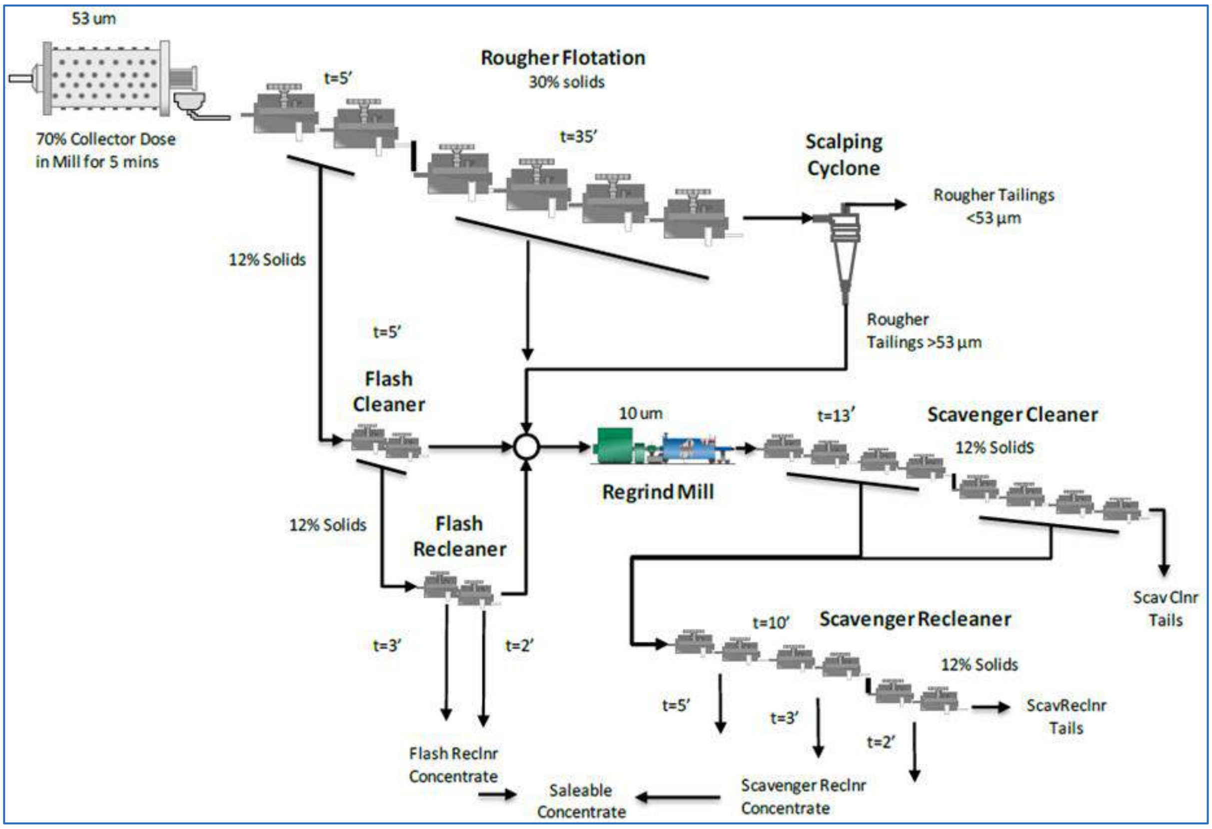


Image courtesy XPS, 2015.

Table 13.14 Flotation Results – IFS4 Circuit

Composite		Final Concentrate					Tail	Feed
		Mass%	% Cu	Rec Cu (%)	$\%SiO_2$	%Fe	% Cu	% Cu
6A	DC	8.53	39.0	88.3	14.6	16.3	0.48	3.76
	90:10 H: S	8.75	37.2	88.7	6.13	22.9	0.45	3.58
	Hypo	8.98	35.7	90.0	4.92	23.4	0.40	3.56
	Super	5.62	48.5	75.3	14.5	8.47	0.95	3.62
6B	DC	8.14	37.0	92.3	7.62	22.7	0.28	3.26
	Hypo	6.29	44.5	91.9	10.6	15.4	0.26	3.05
	Super	5.96	46.5	69.4	15.8	10.6	1.30	3.99
15-year Comp		7.34	39.0	88.1	11.0	17.8	0.42	3.25

In the above tests the 6A supergene rougher flotation stage was slightly acidic and was corrected to pH=7 using lime. A repeat test was conducted in which no lime was added and rougher flotation proceeded at natural pH. These results are summarised in Table 13.15.

Table 13.15 Repeat of 6A Supergene Testing – no pH Adjustment to Rougher Flotation

Composite		Final Concentrate					Tail	Feed
		Mass%	% Cu	Rec Cu (%)	$\%SiO_2$	%Fe	% Cu	% Cu
6A	Super	5.49	51.9	76.1	13.6	9.09	0.95	3.74

The lack of lime in the test has improved both grade and recovery for the 6A supergene sample. It is notable that the tailings grades are identical and, in general, these two results using the one sample show that the repeatability of the test is excellent.

The flow sheet was simplified to what is termed the IFS4a configuration by removing the 53 µm scalping of rougher tailings. This was done because the practical implications of conducting this scalping step are not well represented in the test method for the following reasons:

1. Scalping would actually be carried out using cyclones which have poor efficiency compared to screens, and more fines would be sent to regrinding and flotation.
2. Scalping using cyclones would also result in a loss of some of the oversize to overflow due to inefficiency.
3. An alternative to cyclone scalping of the tailings would be to grind finer before the roughers.
4. In the IFS4 circuits an average of 45% of the plant feed needs to be ground down to 10 µm with the hypogene and composite samples and about 36% with the supergene samples. These proportions compare with 25% and 21% respectively for non-scalping circuits like IFS4a.
5. These high regrind mass proportions increase even further with the use of cyclones to do the scalping.

The complexity of scalping was removed from the design and testwork was repeated to reflect the recommended PFS circuit. The IFS4a circuit is shown in Figure 13.11.

Figure 13.11 XPS IFS4a Flow Sheet – Basis of the Kamoa 2017 PFS

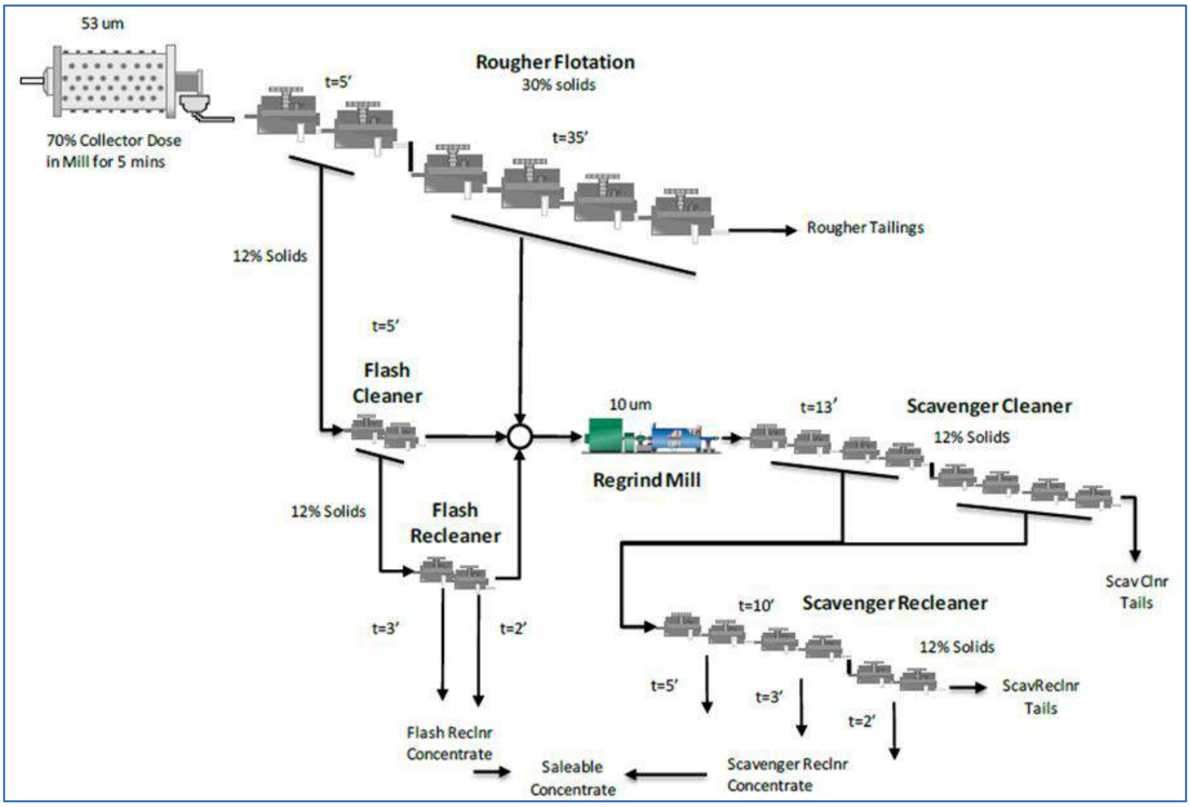


Image courtesy XPS, 2015.

All the tests were repeated with the IFS4a circuit and the results are shown in Table 13.16.

Table 13.16 Flotation Results – IFS4a Circuit

Composite		Final Concentrate					Tail	Feed
		Mass%	**% Cu**	**Rec Cu (%)**	**%SiO_2**	**%Fe**	**% Cu**	**% Cu**
6A	DC	7.80	41.4	86.2	11.1	16.8	0.56	3.74
	90:10 H: S	8.33	37.0	85.4	6.34	22.0	0.58	3.61
	Hypogene	8.48	36.0	86.1	4.00	21.0	0.54	3.54
	Supergene	5.25	53.5	72.3	13.5	13.4	1.14	3.89
6B	DC	8.07	35.4	89.2	9.45	21.3	0.37	3.20
	Hypogene	7.17	35.5	86.9	19.2	13.5	0.41	2.93
	Supergene	6.02	41.2	65.3	19.3	9.65	1.40	3.80

Both the IFS4 and IFS4a tests have been included in this Report to demonstrate the consistency of the test methods being used and to show the sensitivity of copper recovery to the amount of fine grinding employed.

On average across the six test samples, the IFS4a flow sheet loses 3% Cu recovery compared to the IFS4 circuit. The recovery loss will be traded off against the additional power requirements and CAPEX for milling during the FS so that the most economically efficient flow sheet can be selected. However, for the Kamoa 2017 PFS it has been assumed that the benefits of the simpler IFS4a circuit outweigh the losses.

The IFS4a copper concentrate grade and recovery data from Table 13.16 has been plotted in Figure 13.12.

Figure 13.12 Recovery vs Grade Plot for Phase 6 IFS4a Comparative Flotation Tests

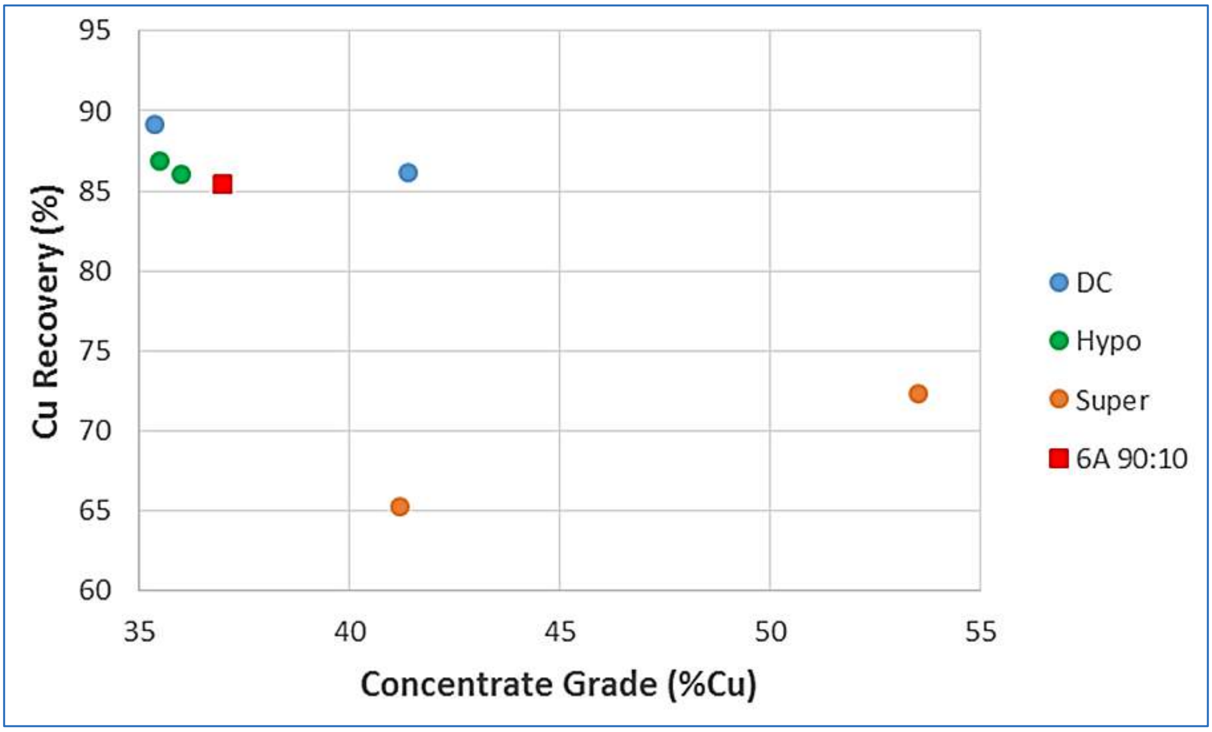


Image courtesy of Amec Foster Wheeler, 2016.

As expected, hypogene samples generate relatively low concentrate grades with good recoveries. The supergene samples generate much higher-grade concentrates but at a significant recovery penalty. The recovery loss is due to copper being present in non-sulphide copper minerals.

13.2.5.3 Copper Recovery vs Head Grade Model

To allow the prediction of copper recovery in the block model (mine planning) it is usually necessary to develop a model relating copper recovery to head grade. The recovery model from the previous Technical Report is presented in Figure 13.13, together with the performance seen in the Phase 6 IFS4a tests.

Figure 13.13 Old Copper Recovery Model (TR 2013)

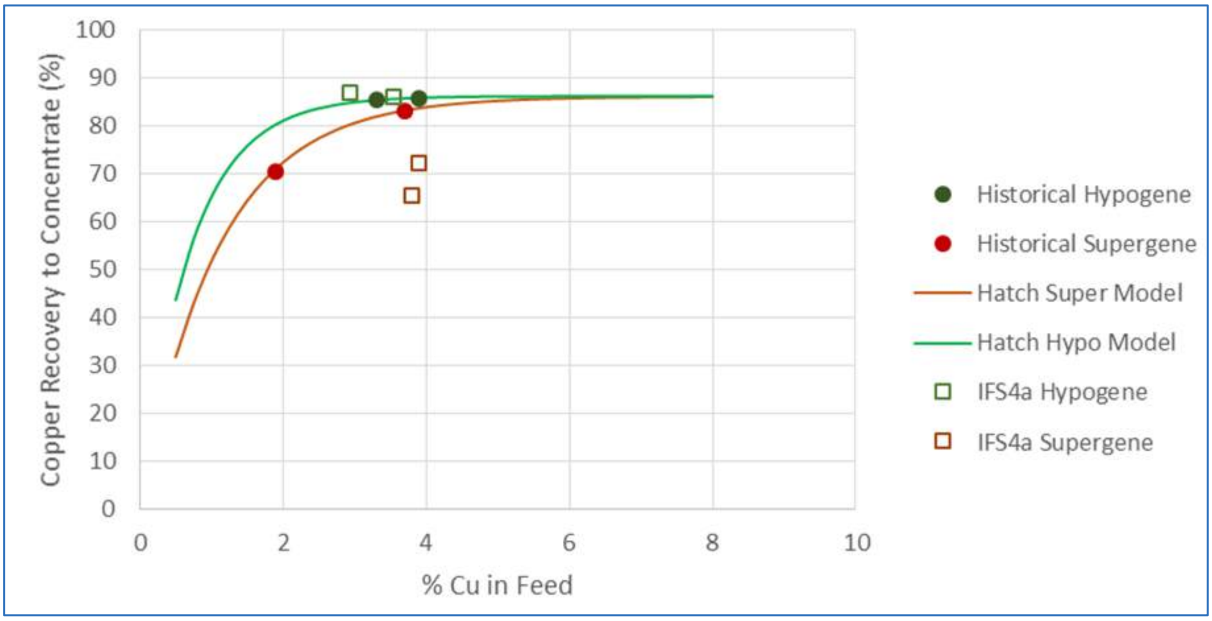


Image courtesy of Amec Foster Wheeler, 2016.

The Phase 6 Hypogene results conform reasonably to the old model, but the supergene response does not. To incorporate the Phase 6 results into the design and planning calculations, improved recovery models are required. In the PEA (2012) a model was developed based on non-floating copper and this has been revived and updated to match the Phase 6 results. As can be seen in Figure 13.14, the new model better represents the Phase 6 results. The new hypogene results were also modelled with less recovery drop-off below 3% Cu.

Figure 13.14 Updated Recovery Models based on PFS Testing

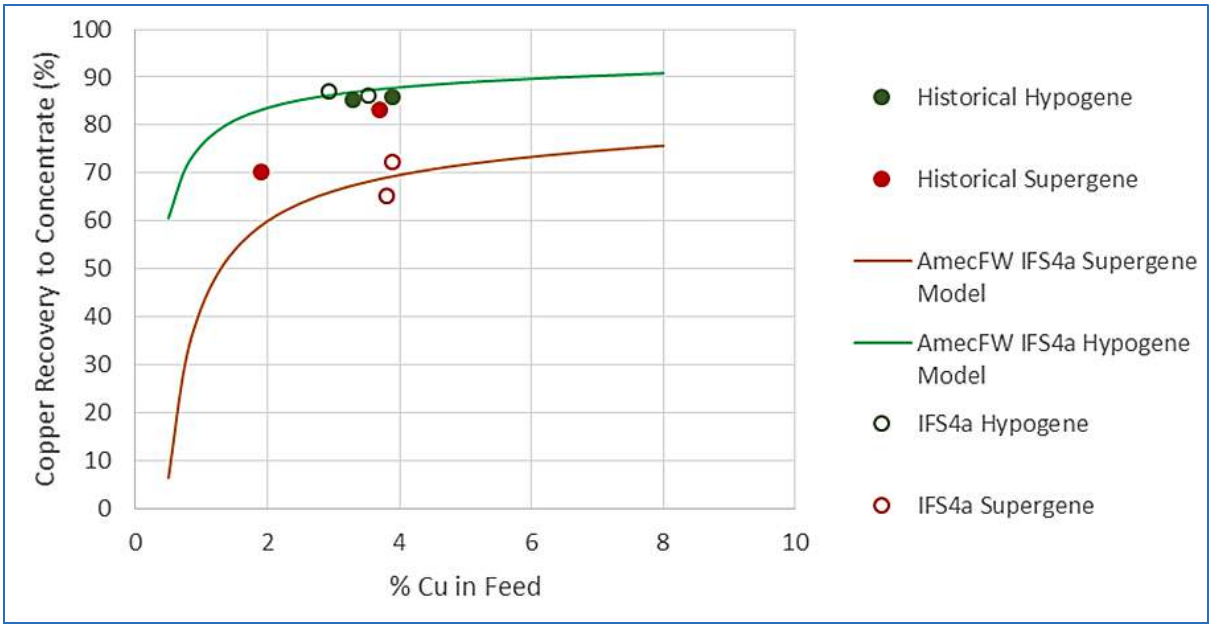


Image courtesy of Amec Foster Wheeler, 2016.

Compared to past models, the new model predicts similar recoveries from hypogene and much lower recoveries from supergene. The lower recoveries for supergene are in line with the test results and are partially the result of high variability in the composition of supergene samples from one test phase to the next. Given that the Kamoa 2017 PFS ore schedule includes the supergene composite samples tested in Phase 6, the modelled recovery reductions are valid.

Supergene Recovery Variability

It is clear from Figure 13.14 above that supergene recovery is not well defined when it is necessary to rely on a single dependency, in this case the head grade of copper. There will be a recovery relationship with head grade, but the analysis shows that the recovery is more dependent upon the proportion of the copper that is not floatable than the grade of copper in the feed.

The block model contains acid soluble copper (ASCu) information, which allows copper recovery predictions to be made for a subset of the supergene mineralisation type. It is only necessary, at this stage of the project, to modify recovery in mineralised zones where the supergene classification is the result of surface oxidation. It is not necessary if it is classified as supergene due to alteration at depth from fluid originating from the sandstone beneath the mineralised zone. Recovery from all “deep” supergene is calculated using the hypogene recovery formula.

In addition, in some intersections the surface oxidation has not been severe enough to increase the proportion of ASCu above the threshold normally seen in hypogene samples, which is in the range of 5% to 15% (it is thought that the ultra-fine component of the sulphide mineralisation, especially chalcocite, is dissolving during the ASCu determination, but this is yet to be confirmed). Consequently, an alternative recovery calculation is only applied for near surface supergene having greater than 15% of total copper being ASCu.

For the Phase 6 testwork on hypogene and supergene samples, the relationship between floatable copper in the feed (as mineralogically defined using QEMScan Analysis) and copper recovery to concentrate is shown in Figure 13.15.

Figure 13.15 Prediction of Copper Recovery Using Mineralogy

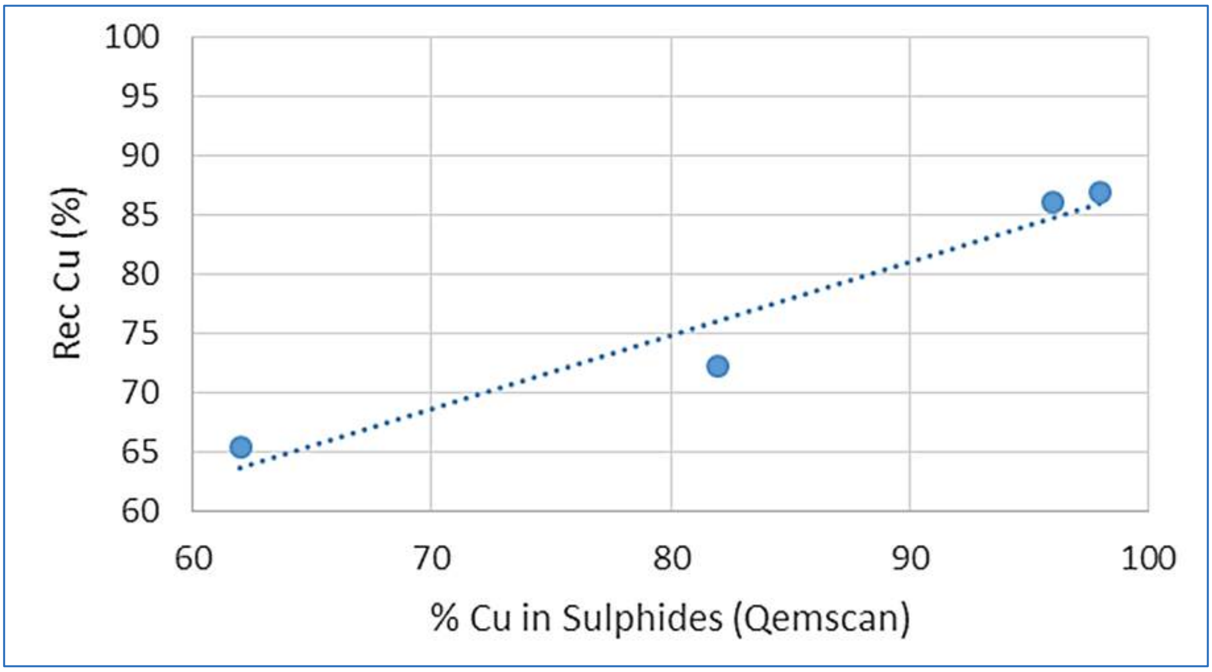


Image courtesy of Amec Foster Wheeler, 2016.

This strong relationship between recoverable copper and copper in sulphides is expected. Almost all oxide copper minerals, together with native copper, are not readily floated in a standard copper sulphide flotation chemical environment, which uses relatively low concentrations of selective collectors.

13.2.5.4 Phase 6 Testwork – Signature Plot XPS

A signature plot is used to design and select an IsaMill by determining the specific energy requirement for the regrind duty. It is necessary to generate 18 kg of representative IsaMill feed material to conduct the test, and this was achieved by performing 39 modified IFS4a (2 kg) flotation tests. As the full IFS4a flow sheet includes regrinding, it was necessary to truncate the tests ahead of the regrinding stage at each point. The test format is shown in Figure 13.16.

Figure 13.16 Truncated XPS IFS4a Circuit

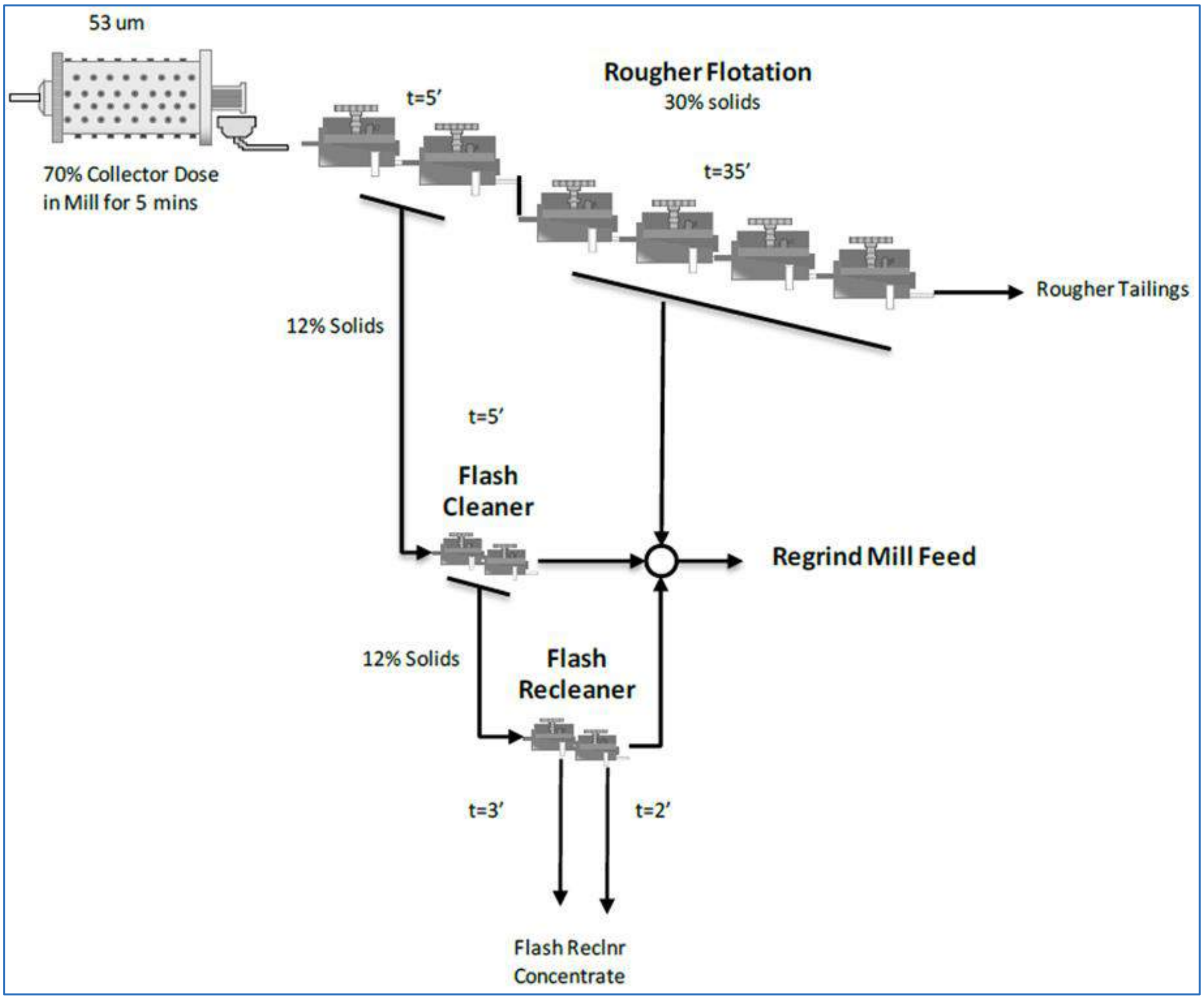


Image courtesy XPS, 2015.

The 6A signature plot composite was prepared separately from the other composites and contained 4.35% Cu. The Cu:S is 1.37 compared to 1.66 for the 6A DC sample indicating a greater proportion of chalcopyrite in the copper mineral suite of the new composite.

Although the rougher feed was ground to a P_{80} of 53 µm, the regrind mill feed was much finer with a P_{80} of 34 µm. The regrind feed contained 56% of material finer than 10 µm and 4% of material coarser than 100 µm. The regrind feed represented 30.8% of the new feed by mass, higher than the 24% of mass estimated for the 6A DC composite. The higher mass is partially driven by the higher feed grade and also increases because the Cu:S ratio is lower.

The IsaMill feed grade was relatively low at 6.6% Cu and contained almost half (47%) of the copper in the test feed. The SG of IsaMill feed was measured at 2.98. Xstrata set the IsaMill feed percentage solids at 41% to avoid viscosity problems potentially associated with a 10 µm regrind target.

The IsaMill feed sample was passed through the M4 IsaMill test unit multiple times, and samples were taken of the product at each pass. The resulting signature plot is shown in Figure 13.17.

Figure 13.17 IsaMill Signature Plot

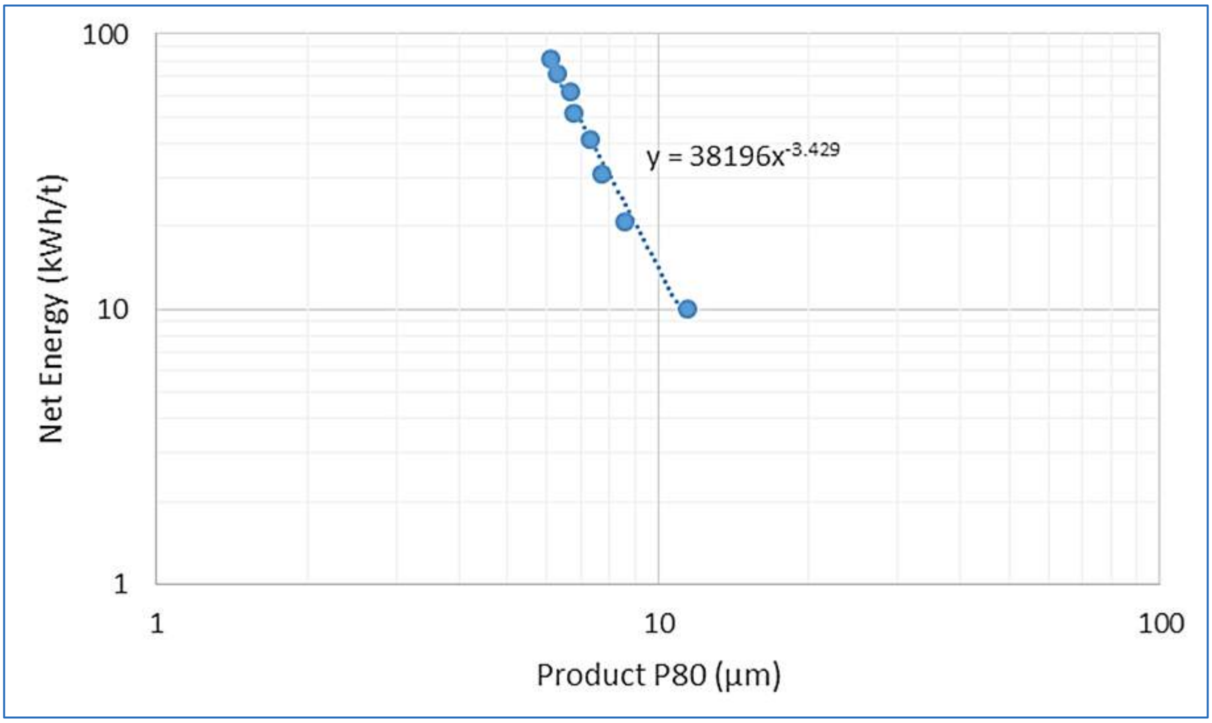


Image courtesy of Amec Foster Wheeler, 2016.

This result is based on the sample tested, and the specific grinding energy requirement for other feeds will be dependent upon the P_{80} of the regrind feed and the mineralogy of the feed. An analysis of the various Phase 6 tests showed that these factors, together with the mass pull to be reground, vary considerably as summarised in Figure 13.18.

Figure 13.18 Phase 6 Regrind Feed Variability

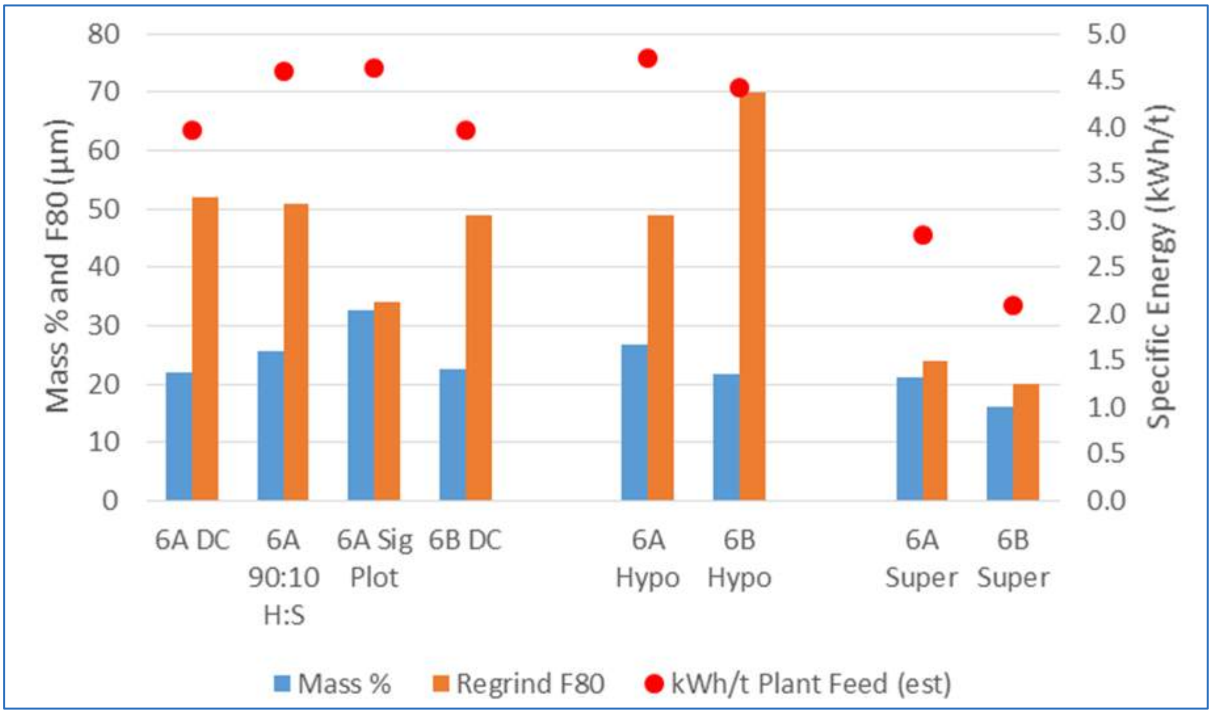


Image courtesy of Amec Foster Wheeler, 2016.

Interestingly, across the four development composites and two hypogene samples the energy per tonne of plant feed is somewhat independent of the test. This is because low mass pulls tend to have coarse particle sizes while high mass pulls are finer. From the Figure 13.18 data, a regrind power selection of 5 kWh per tonne of plant feed should be sufficient to provide regrind capability in the Kamoa 2017 PFS circuit.

The supergene composites only require 3 kWh per tonne of plant feed, but are not planned to be mined or processed in isolation and will not be subjected to overgrinding.

13.2.5.5 Kamoa Phase 6 Variability Testwork

A programme of variability testwork has been planned for Kamoa using the samples indicated in Figure 13.19 together with the Year 0 to 15 PFS mining areas.

Figure 13.19 Planned Phase 6 Variability Samples

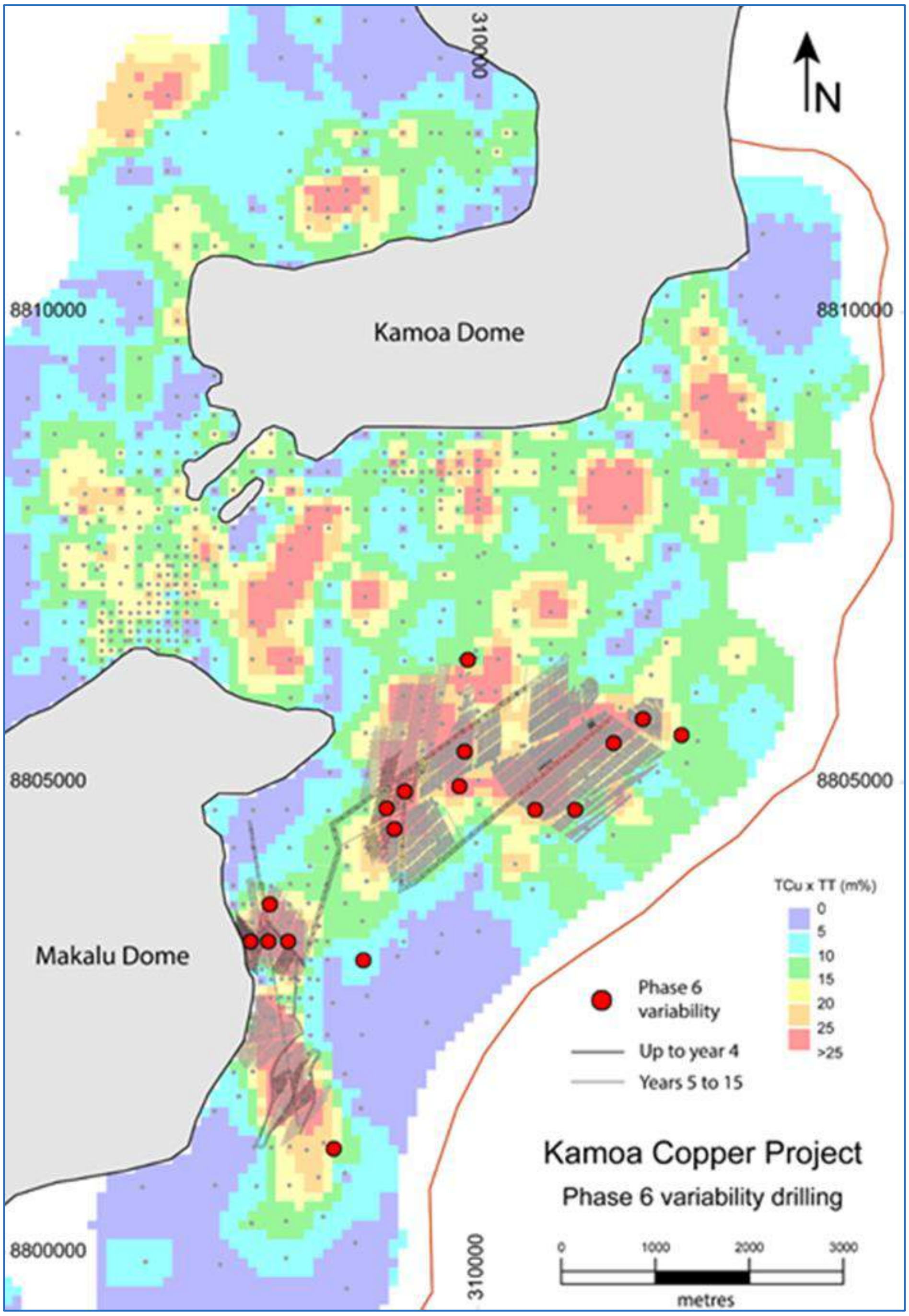


Figure provided by Ivanhoe, 2016.

The variability sample selections provide good spatial representation of plant feed during the proposed Kansoko mine plan period. However, due to shifting project priorities these samples remain in refrigeration ready to be tested in the future.

13.2.5.6 Kamoa Mineralogy

The Kamoa copper sulphide mineralisation exists in two basic modes regardless of copper sulphide mineral. Coarse copper sulphides, some in the centimetre size range, are clearly visible in the core. Many intermediate sized copper mineral grains are usually visible but any that are clearly distinguishable can be considered coarse. The second mode of occurrence is a pervasive "fog" of ultrafine copper sulphides throughout the matrix.

In the image below (Figure 13.20) can be seen a 2 cm-wide white clast within the grey diamictite matrix, against which chalcopyrite has "mantled" during the sulphide deposition phase. In the surrounding rock matrix there are smaller mantled clasts and visible blebs of chalcopyrite (and other sulphides). What cannot be seen in the photograph is the dispersion of 1 to 10 µm (0.0001 to 0.001 cm) copper sulphides present throughout the grey matrix.

Figure 13.20 Typical Kamoa Hypogene Mineralisation in Diamictite

Figure Courtesy Amec Foster Wheeler, 2011.

QEMScan, an automated particle analysis system, has been used to reveal the fine mineralogical detail of Kamoa samples. Two rougher flotation tests were conducted on the 6A development composite by XPS, in which six concentrates were collected sequentially after grinding the samples to P_{80} 53 µm and 38 µm respectively. The QEMScan analysis was used to derive the proportion of liberated copper in each of the concentrates, and the results are summarised in Figure 13.21.

Figure 13.21 Copper Sulphide Liberation in Rougher Flotation

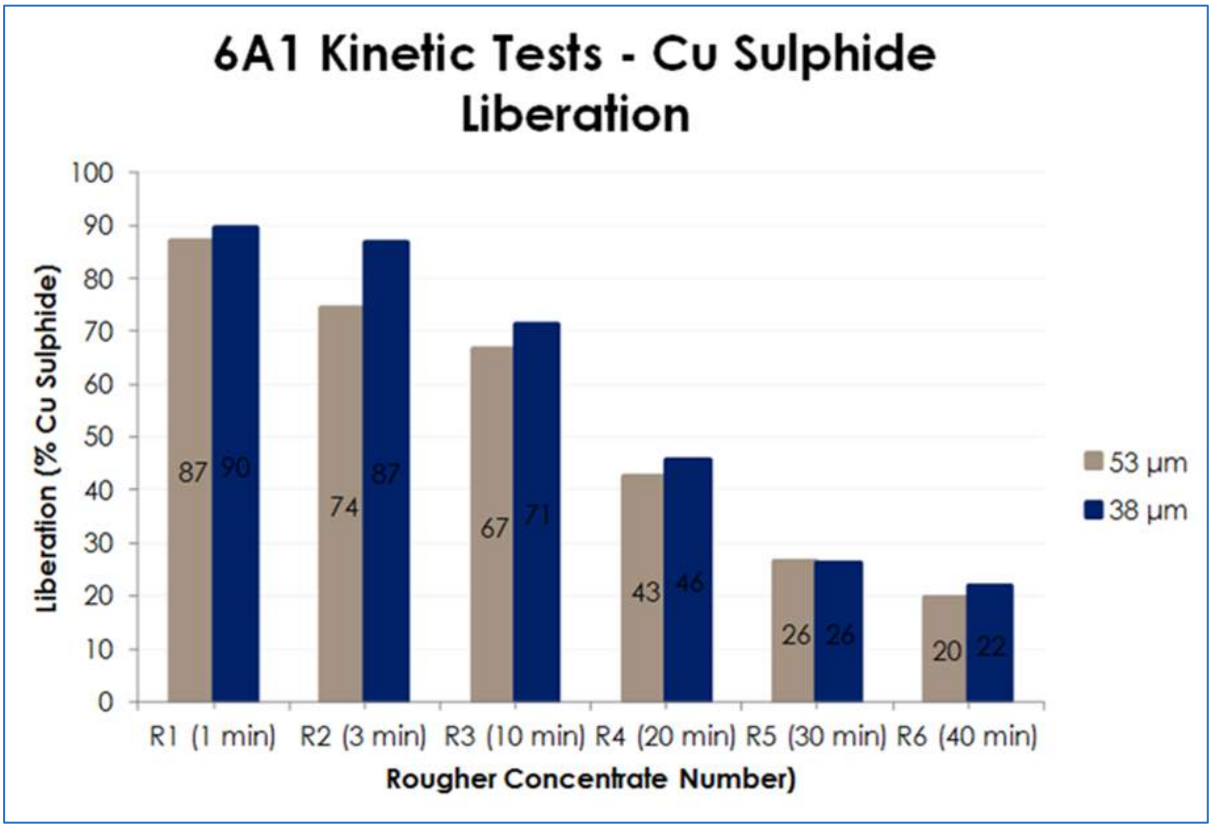


Image courtesy XPS, 2015.

The highly-liberated copper sulphides are floated preferentially while the poorly liberated sulphides float towards the end of the test. It is also clear that at the finer grind size (+38 µm) the overall liberation level is higher than in the 53 µm test.

Copper sulphide morphology in all Kamoa and Kansoko samples is consistent in that the minerals are always present as both very coarse and very fine grains. The large proportion of copper in fine sulphides is the reason for the strong liberation effect of grinding (measured using QEMScan, XPS Laboratory) as shown in Figure 13.22.

Figure 13.22 Phase 6 Hypogene Composite Liberation Analysis

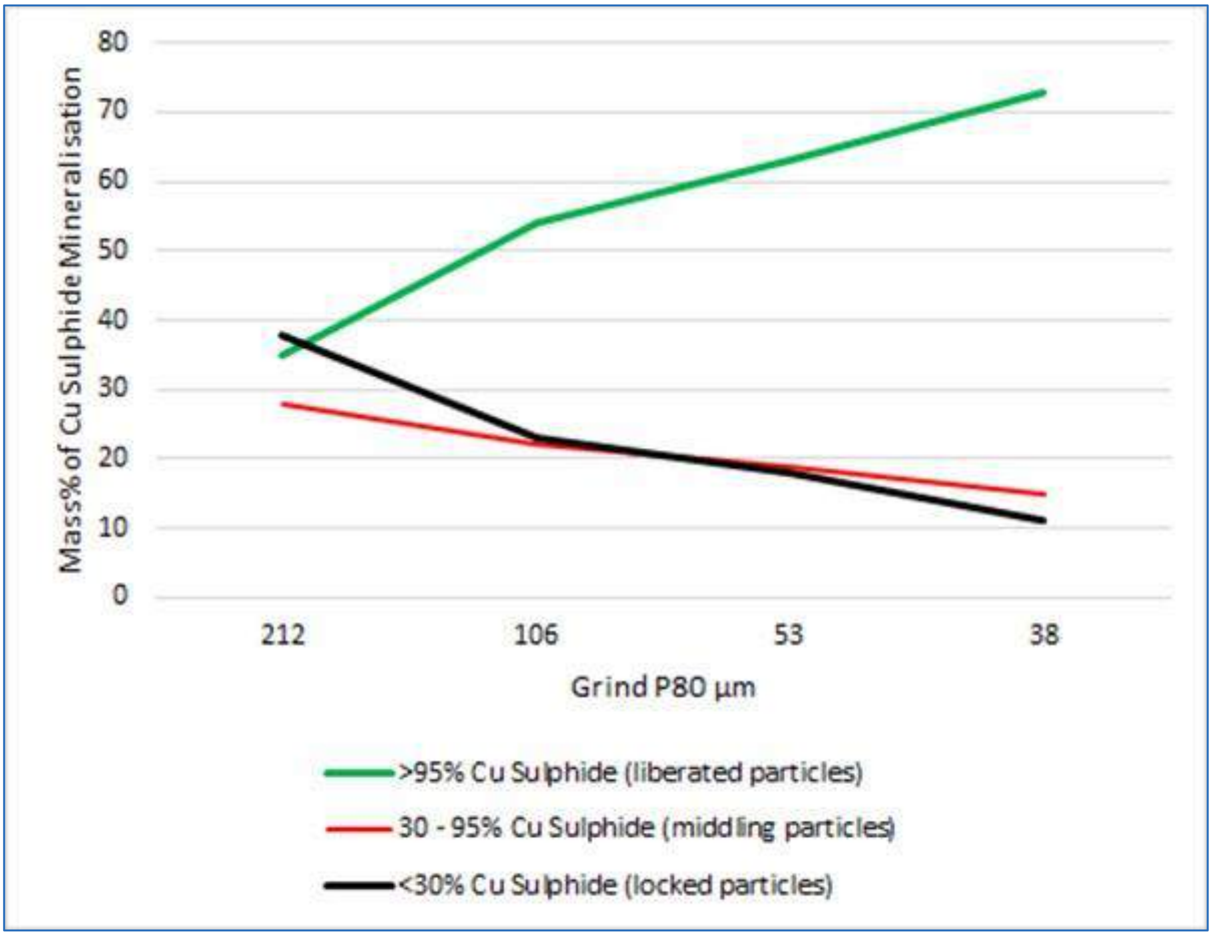


Image courtesy Amec Foster Wheeler, 2018: from XPS data, 2014.

At the fine grind P_{80} of 38 µm, 27% of the copper sulphides remain unliberated. Almost half of these are in the very poor grade "locked" class and are generally unavailable to recover in flotation. If locked particles are recovered they rarely survive the cleaning process and are rejected to tails at some point in the flow sheet.

QEMScan also generates particle mineral maps and is able to group both minerals and particles to assist in visual examination. Figure 13.23 is a liberation grid showing particle sizes (vertical) and liberation classes (horizontal). Minerals have been grouped into six important categories rather than the tens or even hundreds of minerals that are identified in the original analysis. In these images there is very little "Other Cu" which includes minerals like malachite and native copper. The main copper mineral class is CuFeS (yellow) which consists of grouped chalcopyrite and bornite. The other copper mineral class is CuS (red) which consists of grouped chalcocite and covellite. Note that the CuFeS and CuS classes are both targets for recovery; thus the definition of liberation is based on a further grouping of these two classes.

Figure 13.23 Combined Copper Sulphides Liberation Map – Rougher Concentrates R3 to R6

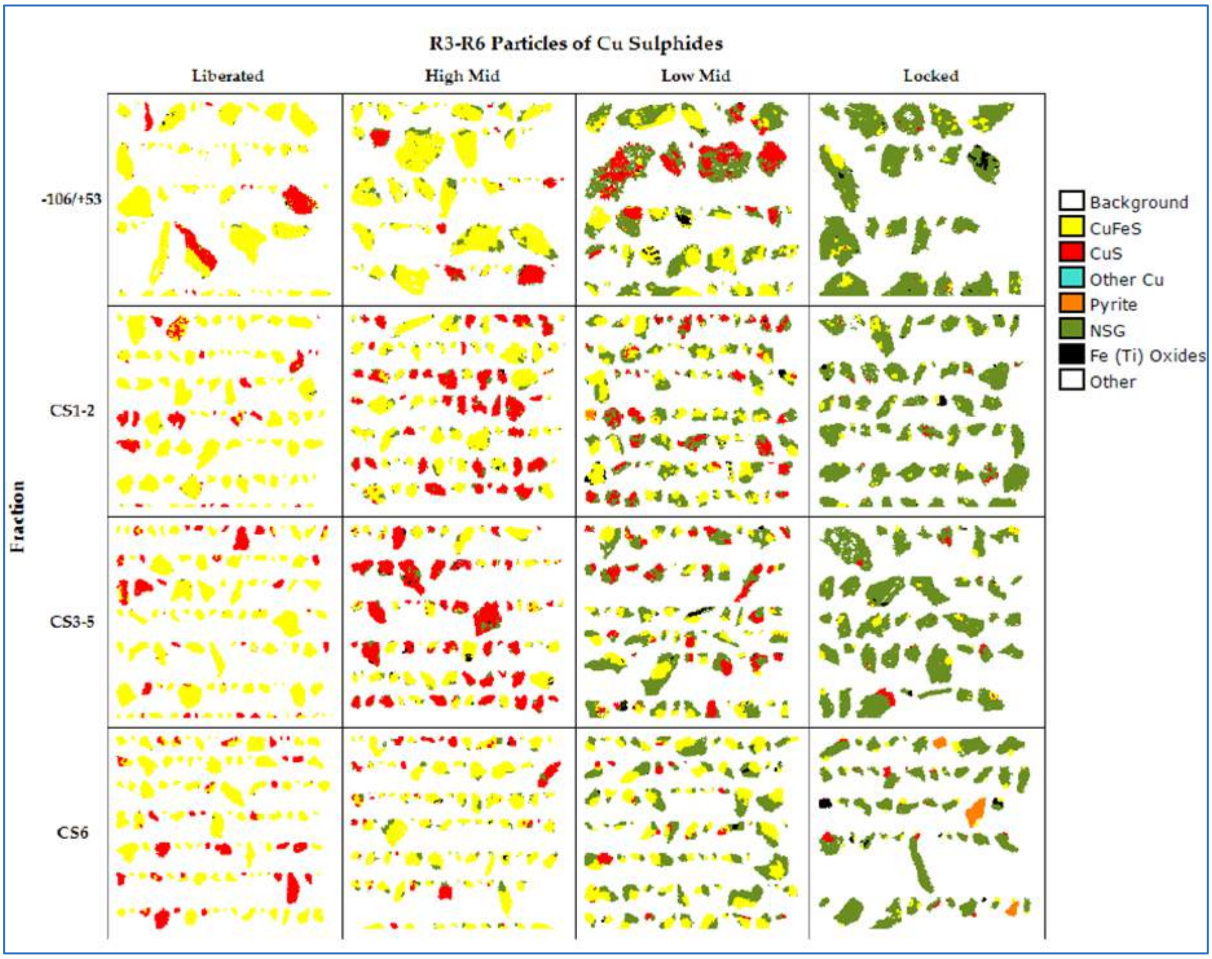


Image courtesy XPS, 2015.

It is clear that even in the CS6 (cyclosizer cone 6 fraction, particle size about 4 µm) there is a large amount of the copper held in poorly-liberated particles. The copper sulphide phases in the CS6 particles are typically 1 to 3 µm. This poor liberation of fine sulphides is a characteristic pervading the entire Kamoa mineralised zone and has driven the fine grinding component of the flow sheet development.

All particles in Figure 13.23 above have been floated or transported to the concentrate by entrainment with the froth water. All that is needed for a particle to float is a small exposure of copper sulphide at the surface and the “low Mid” and “Locked” particles in the image shows that this is generally the case.

The pervasive fine copper sulphides cause large amounts of attached silicates to be recovered in rougher flotation and this leads to the high rougher mass pull values (20% to 40%) typical in the test programs. At coarse grinds, such as 150 µm P_{80}, large silicate particles invariably have exposed fine copper sulphides on the surface and are able to float.

The fine sulphides also mean that regardless of the rougher flotation size it is necessary to regrind middlings material to ultra-fine sizes to achieve low silicate levels in final concentrates. Testing has shown the concentrate quality to be sensitive to regrind P_{80}, with 15 µm producing poor concentrates and 10 µm generally producing acceptable concentrates.

Another notable aspect of Figure 13.23 above is the general absence of pyrite. It is only at the finest size that pyrite appears, and this indicates that composites or binary particles containing both pyrite and copper sulphides are scarce.

The major source of copper loss in flotation has been examined by QEMScan analysis of the rougher tailings. The liberation map for Rougher tails is shown in Figure 13.24.

Figure 13.24 Combined Copper Sulphides Liberation Map – Rougher Tails

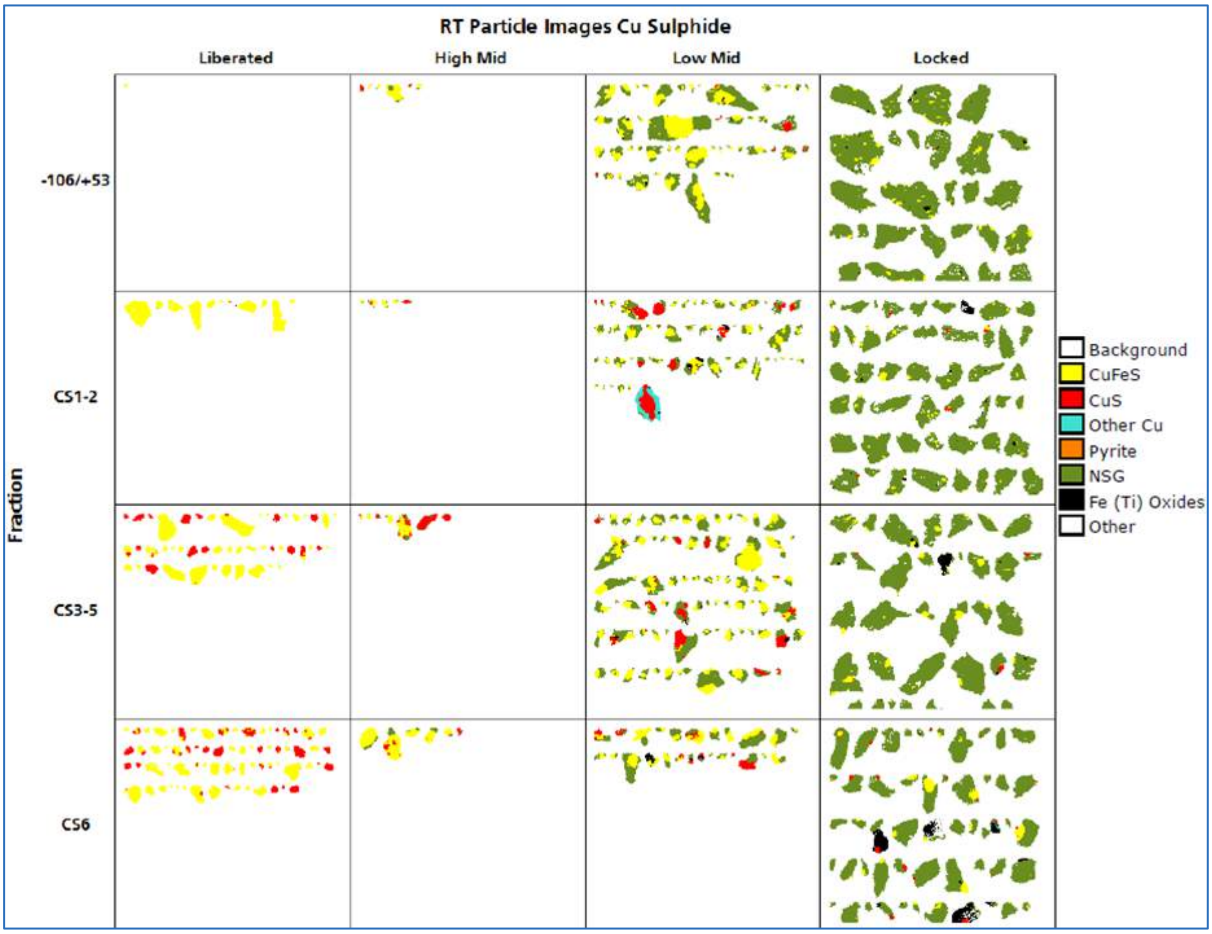


Image courtesy of XPS, 2015.

Although there are some fine liberated particles shown as being lost to rougher tailings it is not possible, from this image alone, to determine how significant these few particles are in terms of copper recovery loss. Typically, the majority of lost copper will be in the Low-Mid and the locked classes, simply because they represent the greatest mass proportion.

Many of the low-mid particles may have floated with longer roughing time, but typically they report to tails because the surface of the sulphides is passivated or the actual amount of sulphide exposure is low (it must be remembered that these images are particle cross-sections and the real state of mineral exposure in three dimensions is unknown).

As can be seen in Figure 13.25, regardless of the size fraction, the lost copper sulphides are in phases that have average grain sizes of less than 10 µm.

Figure 13.25 Copper Sulphide Phase Size in Rougher Tailings

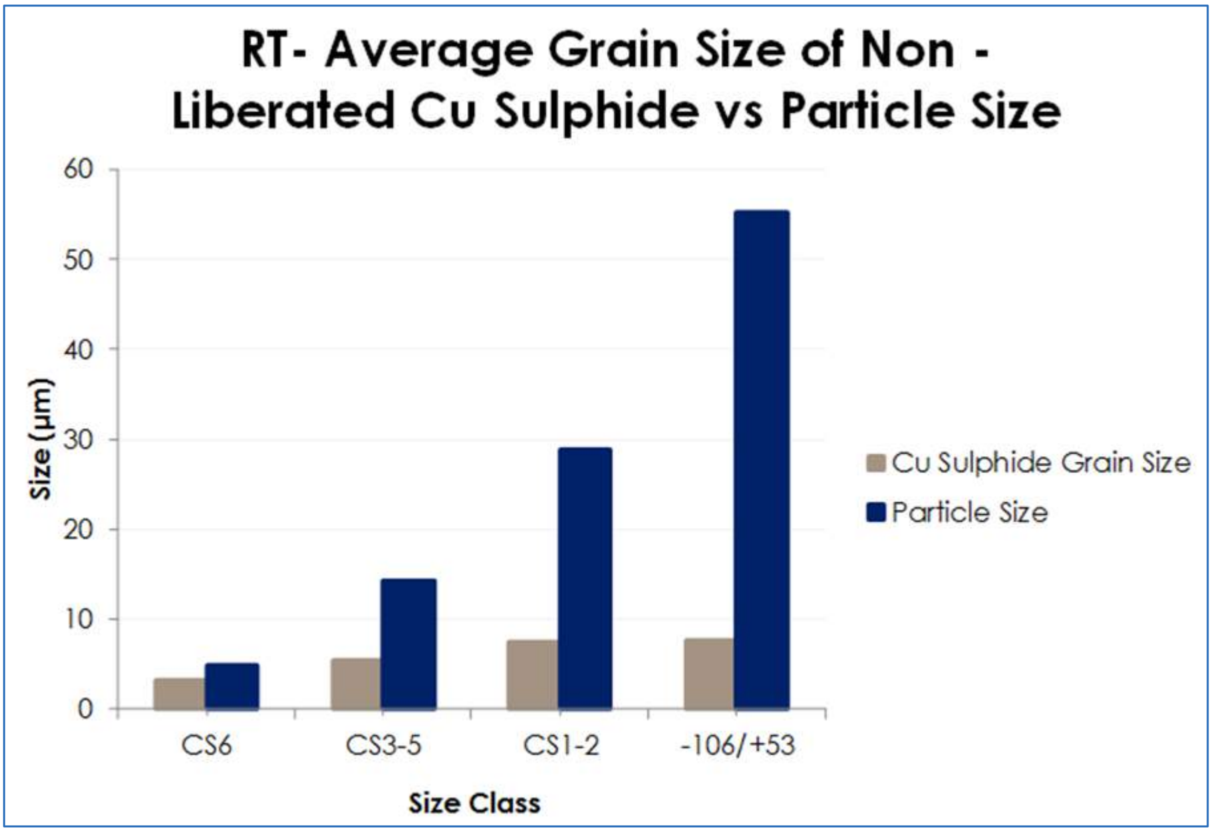


Image courtesy of XPS, 2015.

The flotation testwork has progressed to a point where recoveries in rougher flotation are typically above 90% and the material lost to tailings is dominated by ultra-fine locked copper sulphides. It has also progressed to the point where the need for ultrafine regrinding has been confirmed and high recoveries are being achieved at high concentrate grades.

13.3 Testwork on Kakula Resource

The initial metallurgical testwork, on the Kakula resource, was conducted during 2016–2017, at Zijin laboratories in China and XPS in Canada, under management of Kamoa Copper SA. Following the successful preliminary testing, additional drill core material was tested as part of the Kakula PFS campaign, which focussed on flow sheet optimisation as part of the Kakula project PFS.

The PFS testwork campaign (2017–2018) consisted of the following:

- Mineralogy and sample characterisation on a mill feed and a final concentrate sample, conducted by XPS.
- Comminution testing, conducted by Mintek.
- Flotation flow sheet optimisation and preliminary variability testing, conducted by XPS.
- HPGR scoping and pilot plant testing, conducted by ThyssenKrupp, South Africa.
- Concentrate thickening and filtration testwork, conducted by Outotec, Canada.
- Tailing thickening and filtration testwork, conducted by SGS, Canada.
- Bulk material flow testwork, completed by GreenTechnical in South Africa.

13.3.1 Kakula Metallurgical Sample Locations & Descriptions

Refer to Figure 13.26 for an illustration of the positions of each of the drill cores tested during the preliminary and PSF testwork campaigns.

13.3.1.1 Preliminary Flotation Sample

Preliminary flotation testwork, for Kakula, was conducted on three composite samples from six different drill cores. Initially, two drill core samples from early holes, DD996 and DD998, were used for testing. Each of the two samples were tested individually, as well as a 50:50 composite sample of the two cores, referred to as Flotation Composite 1.

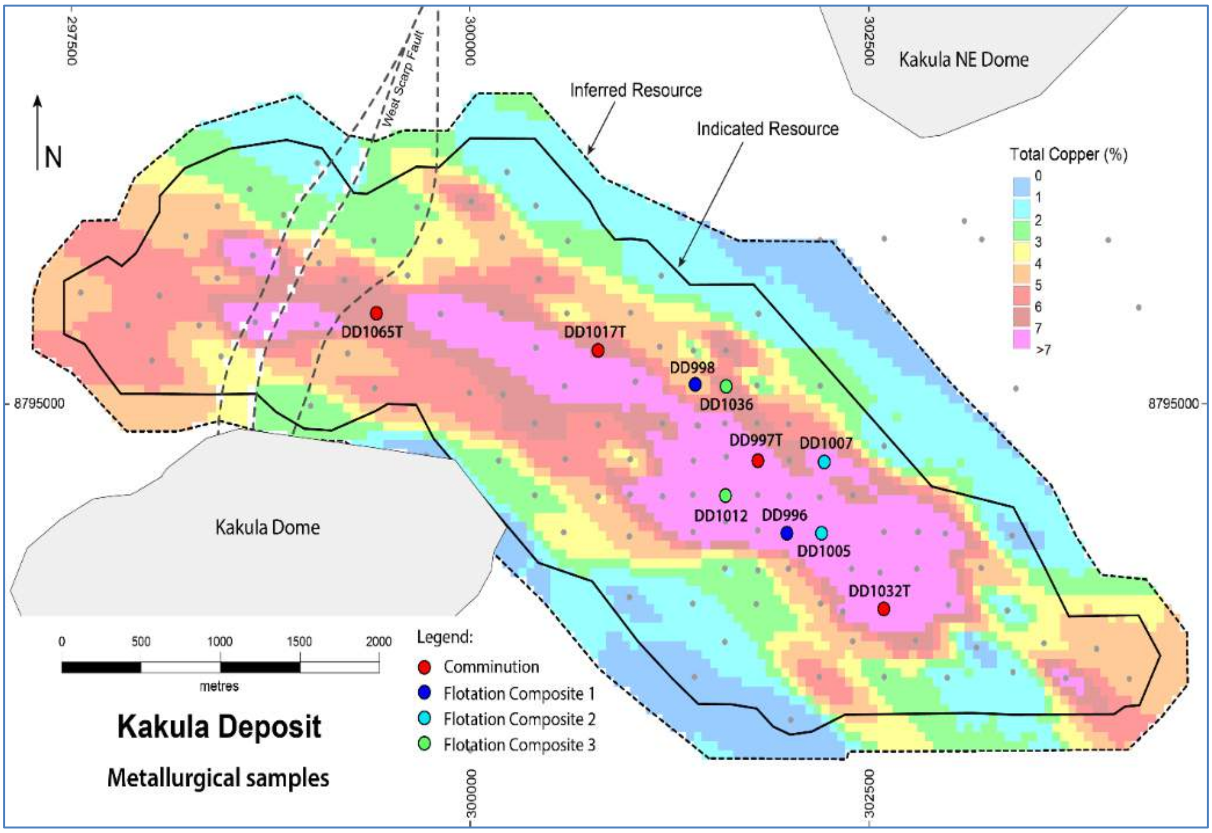
Following successful testing of these early holes, and due to high-grade intercepts consistently achieved at Kakula, additional samples from drillholes DD1005 and DD1007 (Flotation Composite 2) were sent to Zijin laboratories, and DD1012 and DD1036 (Flotation Composite 3) were shipped to XPS to verify metallurgical characteristics of higher-grade samples and to reconfirm if the Kakula material was compatible with the IFS4a flow sheet, as developed during Kamoa Phase 6 testwork campaign.

Head analysis were conducted in triplicates on each of the above flotation composite samples. Refer to Table 13.17 below for a summary of the head analysis results for the various flotation samples.

Table 13.17 Kakula Preliminary Flotation Samples Head Analysis

Sample	Cu %	S %	SiO_2 %	Fe %	Al_2O_3 %	CaO %	MgO %
DD996	4.21	1.19	54.5	4.61	12.1	1.86	4.43
DD998	3.96	1.15	52.7	5.16	10.8	2.62	4.98
Flotation composite 1 (50% DD996: 50% DD998)	4.08	1.20	55.5	5.07	12.7	2.19	4.71
Flotation composite 2 (50% DD1007: 50% DD1005)	8.19	2.00	52.82	4.92	13.24	0.96	3.47
Flotation composite 3 (50% DD1036: 50% DD1012)	8.12	1.95	52.34	4.97	13.27	0.86	3.76

Figure 13.26 Drillhole Location Map for Kakula Metallurgical Samples



13.3.1.2 Kakula PFS Comminution Sample

Four PQ drillhole samples (DD1065T, DD997T, DD1017T, and DD1032T) were selected for comminution testing and the different lithologies (footwall (FW), diamictite (SDT) and siltstone (SSL)), per hole was composited to form the following ten samples: DD1065 SSL, DD1065 SDT, DD997 SDT, DD997 SSL, DD1017 SDT, DD1017 SSL, DD1032 SDT, DD1032 SSL, FW SDT, and FW SST. Remainders from these samples were used for HPGR scoping tests.

A further nine samples from drillholes DD1047W1, DD1084W1, DD1021W2, DD1061W1, DD1070W1, and DD1145W2 were selected for comminution variability testing:

- 5 x individual silstone samples,
- 2 x individual diamictite samples,
- 1 sandstone footwall composite sample, and
- 1 diamictite footwall composite sample.

13.3.1.3 Kakula PFS Flotation Sample

During the PFS campaign a total of 10, ¾ HQ drill cores were selected in order to prepare composite samples, that were representative of the anticipated mining area and mining grades (as guided by the 2016 PEA mining plan), for the various testwork campaigns. Drillholes (DD1017TW1, DD1020TW1, DD1029TW1, DD1032TW1, DD1043TW1, DD1065TW1, DD1075TW1, DD1081TW1, DD1112TW1, and DD997TW1) were used to prepare the PFS flotation master composite sample. Head analysis were conducted in triplicate, on the PFS master composite sample, and is summarised in Table 13.18.

Table 13.18 Kakula PFS Flotation Master Composite Sample Head Analysis

Sample	Cu %	S %	SiO_2 %	Fe %	Al_2O_3 %	CaO %	MgO %	As %
PFS flotation master composite sample	6.13	1.66	56.47	5.16	13.73	1.25	4.10	<0.01

For the core samples listed above, 10 kg of each were kept aside, during sample preparation, and used in the preliminary flotation variability testwork.

13.3.2 Mineralogical Studies

XPS conducted mineralogy work on the Flotation Composite 1 (Kakula FC1) and high-grade Flotation Composite 3 (Kakula FC3) samples during September 2016. The scope of work included bulk modal analysis with Cu deportment, grain size and liberation investigations. The mineralogy of the two Kakula samples were compared to the Kamoa Phase 6 development composite sample (Kamoa 6A1DC).

Further mineralogical investigations were conducted by XPS, during 2017-2018, as part of the PFS flow sheet development. QEMSCAN was used on the Kakula PFS flotation master composite sample (Kakula PFS) to determine the bulk modal mineralogy, average grain size, liberation, and level of locking of sulphide particles in each sample.

Figure 13.27 below summarises the results from the Kakula PFS sample bulk modal analysis, as compared to the Kamoa 6A1DC sample and the Kakula FC3 sample. Refer to Figure 13.28 for a comparison of the combined Cu sulphide grain size distributions between Kamoa 6A1DC and Kakula FC1 samples.

Figure 13.27 Kamoa 6A1DC, Kakula FC3, and Kakula PFS Sample Mineralogy

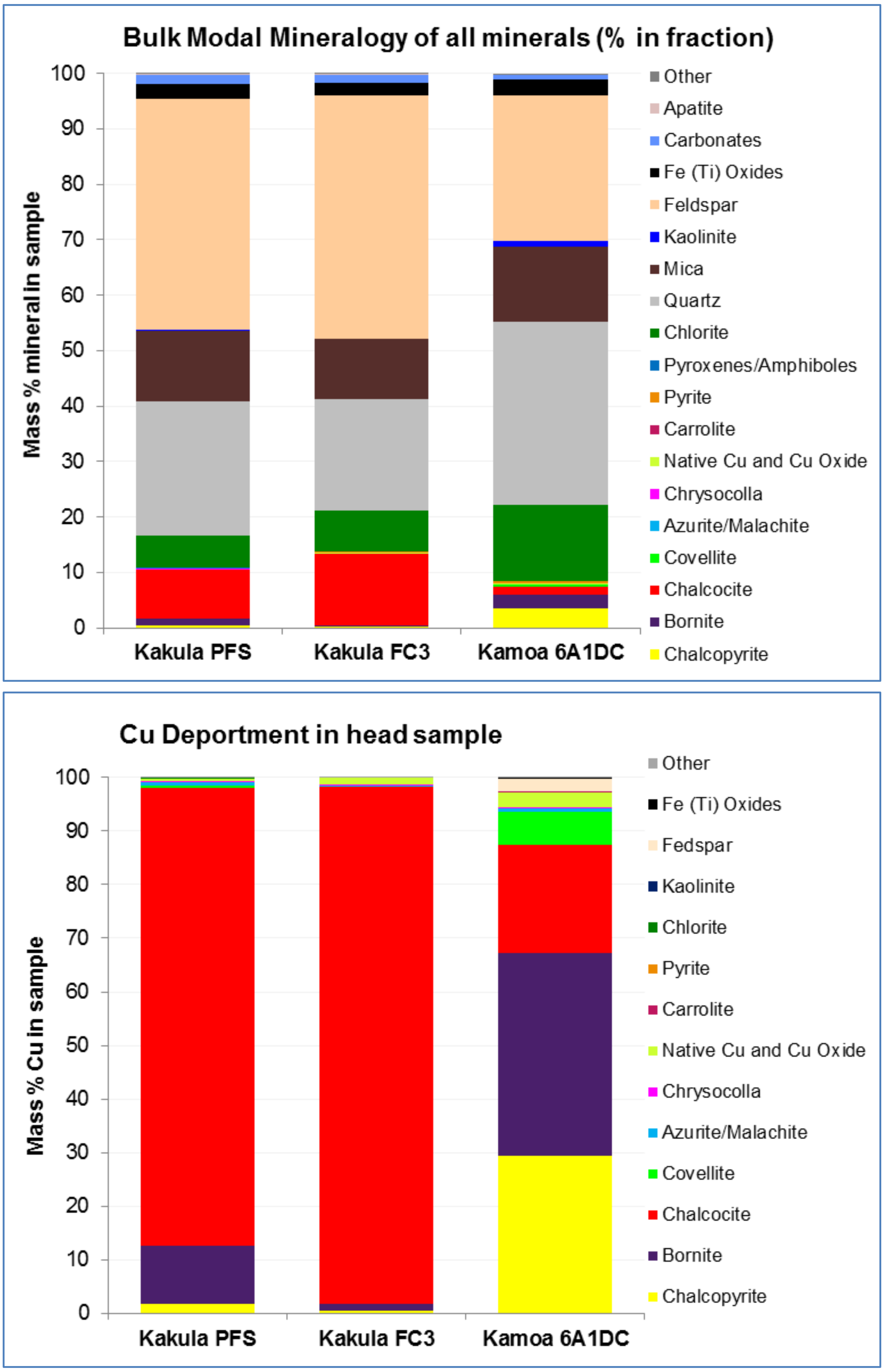


Figure 13.28 Combined Cu Sulphide Grain Size Distribution Comparison Between Kamoa 6ADC and Kakula FC1

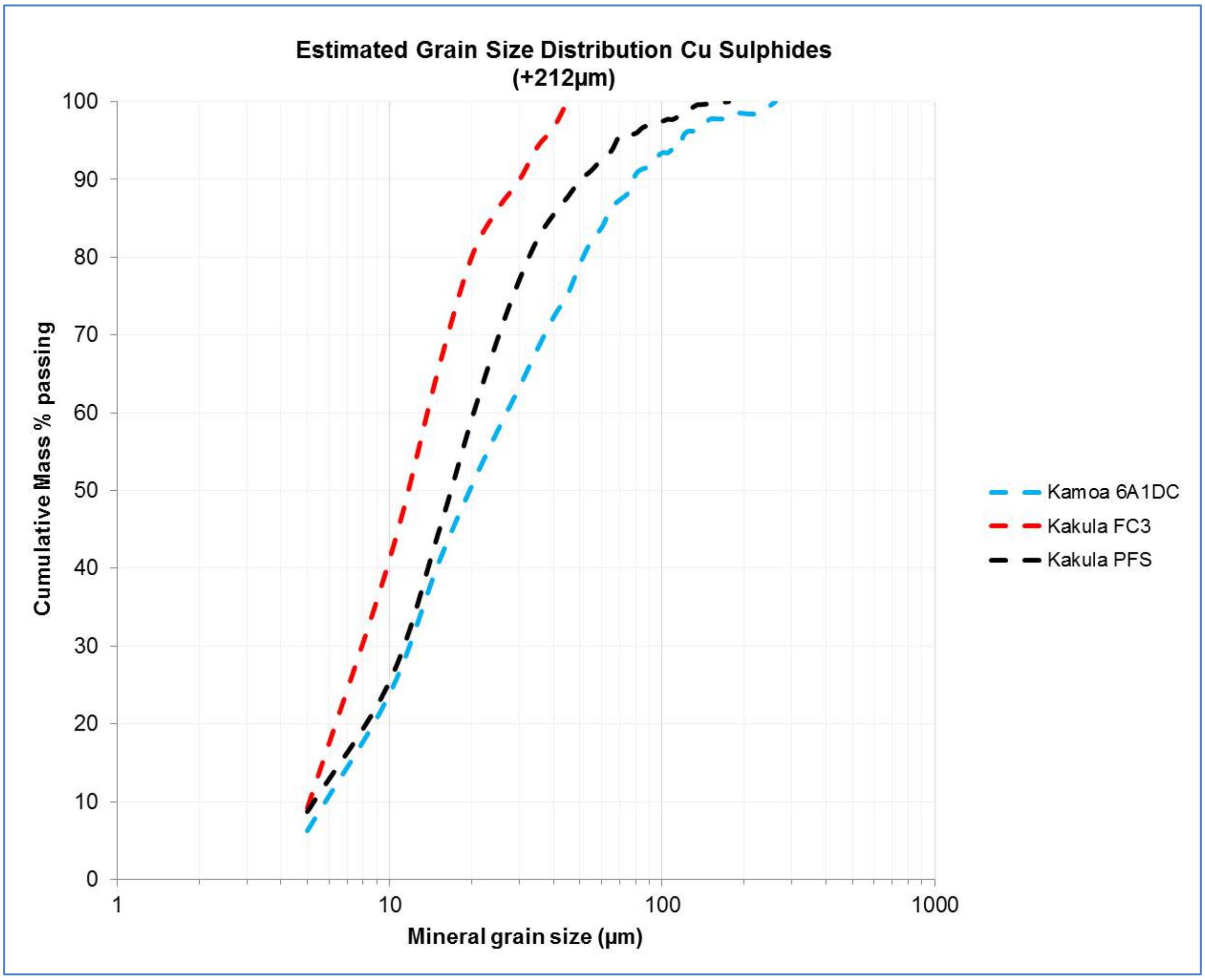


Figure provided by Ivanhoe, 2018.

The following was noted:

- The main Cu sulphide mineral in the Kakula samples was chalcocite, with minor amounts of bornite and covellite. Trace amounts of chalcopyrite was detected with very low amounts of oxides.
- The main gangue minerals were quartz, feldspar, micas and chlorite. The Kakula samples were significantly higher in feldspar when compared. The to the Kamoa 6A1DC sample, but lower in quartz, chlorite and mica.
- Both Kakula ore samples (Kakula FC3 and Kakula PFS) were chalcocite rich, however, the Kakula PFS sample had higher levels of bornite and chalcopyrite compared to the Kakula FC3 sample.
- The average grain size of the Kakula PFS sample sulphide was 33 µm, which was slightly coarser than the Kamoa 6A1DC sample (20 µm). The Kakula FC3 however had a finer grain size of 9 µm, showing variation in the Kakula material grain sizes.

- Although Liberation data at the product size of 80% -220 µm showed that the total of the "liberated plus free" classes is effectively equal for each sample, at approximately 45%, more mineral occurs in the "free" class for the Kakula FC3 sample. There are major differences at the "locked" end of the comparison with the Kakula FC3 sample. Having approximately half the locked Cu of the Kamoa 6A1DC sample.
- The average grain size of the Cu sulphide minerals in the Kakula PFS composite sample was finer than the Kamoa 6A1DC sample at 12 µm, which was consistent with the Kakula FC3 sample (10 µm). Approximately 25% of the Cu sulphide minerals' mass occur in the sub 10 µm ranges, while roughly 8% occurs in the sub 5 µm range.

Chalcocite is a high-tenor mineral that is opaque and dark-grey to black with a metallic lustre. Owing to its very high percentage of contained copper by weight and its capacity to produce a clean, high-grade concentrate, chalcocite is an asset as a dominant copper mineral. Unlike Kamoa, the Kakula deposit has very low bornite, chalcopyrite or other sulphide minerals as seen in Figure 13.29.

Figure 13.29 Comparison of Cu: S between Kamoa and Kakula Mineralisation

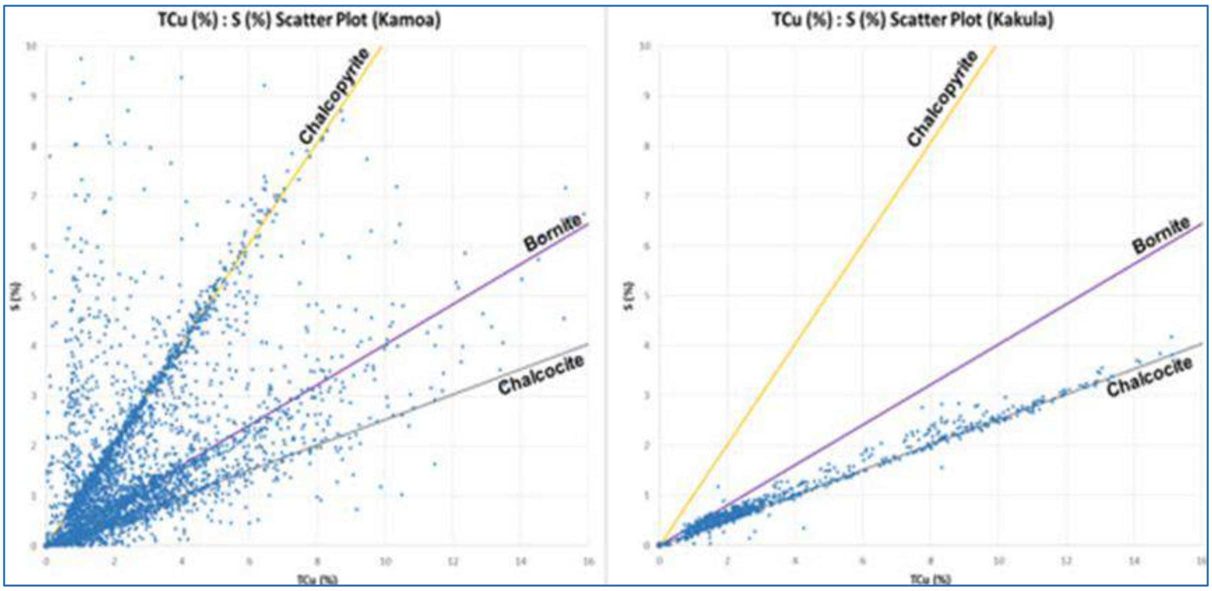


Figure by Ivanhoe, 2016.

This suggests that the Kakula deposit will be relatively easier to treat than Kamoa, and it will be easier to maintain a consistent copper concentrate grade. Mineralogical characteristics are constant across the deposit to date, and there is no indication that they will change significantly. The relatively coarse copper sulphide grain size, the simple mineralogy and the lack of arsenic in feed means that Kakula will generate a valuable concentrate.

13.3.3 Kakula Comminution Testwork

Mintek was contracted by Kamoa Copper SA during 2017 to perform characterisation testwork on diamictite and siltstone samples from four drill cores (DD1065T, DD1017T, DD997T, and DD1032T). Composite samples, of the diamictite footwall and siltstone footwall, were also tested. Comminution parameter variability testing was also completed in 2018, using samples from drillholes DD1047W1, DD1084W1, DD1021W2, DD1061W1, DD1070W1, and DD1145W2.

The scope of work included:

- Uni-axial compressive strength (UCS)
- Bond crushability work index (CWi) and drop weight tests (DWi),
- SAG mill comminution (SMC),
- Bond abrasion index (Ai),
- Bond rod work index (BRWi), and
- Bond ball work index (BBWi) at 75 µm closing screen sizes.

The results of the above tests are summarised in Table 13.19 and Table 13.20.

Table 13.19 Kakula PFS Comminution Parameters Summary

Sample ID	UCS 85th P MPa	DWi kWh/m³	CWi 85th P kWh/t	Ai g	BRWi kWh/t	BBWi at 75 µm kWh/t
DD1017T SDT	69.0	–	10.1	0.02	20.1	15.4
DD1017T SSL	125.4	–	9.8	0.01	22.7	17.1
DD1032T SDT	86.0	–	13.0	0.02	20.7	16.9
DD1032T SSL	77.0	–	10.9	0.01	19.0	15.4
DD1065T SDT	139.4	12.6	12.0	0.01	24.9	18.8
DD1065T SSL	214.3	–	10.7	0.01	24.5	19.1
DD997T SDT	197.9	–	11.8	0.05	20.0	17.6
DD997T SSL	107.7	12.7	11.2	0.03	24.1	19.5
FW SDT 1	81.0	–	13.5	0.06	20.7	17.9
FW SST 1	154.8	–	12.4	0.32	16.1	17.8
DD1047W1 SDT	–	9.8	11.59	0.02	–	18.91
DD1084W1 SDT	–	10.1	10.92	0.02	–	18.91
DD1021W2 SSL	–	10.7	15.61	0.03	–	19.65
DD1047W1 SSL	–	10.1	12.90	0.02	–	20.28
DD1061W1 SSL	–	12.1	17.08	0.02	–	16.08
DD1070W1 SSL	–	11.3	12.91	0.11	–	17.92
DD1145W2 SSL	–	6.1	8.65	0.11	–	14.23
FW SDT 2	–	7.9	9.46	0.05	–	18.31
FW SST 2	–	7.4	14.29	0.38	–	18.00

Table 13.20 Kakula PFS SMC Parameters Summary

Sample ID	M_{ia} kWh/t	M_{ih} kWh/t	M_{ic} kWh/t	ta	A	b	Axb
DD997T SSL	29.7	25.0	12.9	0.21	75.4	0.31	23.4
DD1065T SDT	30.6	25.7	13.3	0.21	80.8	0.28	22.6
DD1047W1 SDT	24.8	19.8	10.2	0.26	72.9	0.40	29.2
DD1084W1 SDT	25.4	20.3	10.5	0.26	62.1	0.46	28.6
DD1021W2 SSL	26.2	21.3	11.0	0.24	73.5	0.37	27.2
DD1047W1 SSL	24.9	20.0	10.3	0.26	72.9	0.40	29.2
DD1061W1 SSL	27.5	22.9	11.8	0.22	80.2	0.32	25.7
DD1070W1 SSL	27.2	22.3	11.6	0.23	80.3	0.32	25.7
DD1145W2 SSL	17.1	12.4	6.4	0.42	59.6	0.78	46.5
FW SDT 1	21.2	16.2	8.4	0.33	68.0	0.53	36.0
FW SST 2	22.1	16.6	8.6	0.35	75.6	0.46	34.8

Initial PFS CWi testing indicated that the Kakula PFS material was soft with regards to crushing energy requirements – however, observations made during the testing noted the presence of pre-existing cracks in the core which was most likely responsible for the low CWi values measured. During the latest testing, the CWi values averaged 11.3 kWh/t for the diamictite samples, and 13.4 kWh/t for the silstone samples. The diamictite samples compared well to the earlier tested samples, however, the silstone CWi increased from 10.6 kWh/t measured earlier. DWi values averaged 10.0 kWh/m^3 for the diamictite samples, and 10.1 kWh/m^3 for the silstone samples.

The Ai results generally demonstrated low abrasion tendencies for the Kakula material. The Ai measurements averaged 0.02 g for the diamictite samples, and 0.04 g for the silstone samples.

The BRWi results grouped the Kakula material in the hard to very hard classes, while the BBWi testing grouped all the samples in the very hard class. The variability samples indicated that the BBWi values averaged 18.2 kWh/t for the diamictite samples, and 17.5 kWh/t for the silstone samples.

SMC testing also classified the samples tested as very hard, indicating that the Kamoa and Kakula material was highly competent and not amenable to Semi and/or Fully Autogenous Milling. The Axb values ranges from 22.6 to 46.5 (average 29.9). The maximum values are significantly higher compared to the Kamoa Phase 6 samples (17–28).

The Kakula PFS samples tested had similar competency compared to the Kamoa Phase 6 material.

13.3.4 Kakula Preliminary Flotation Testwork

The initial flotation testwork was performed by Zijin laboratories in China, as well as XPS in Canada. Two drill core samples, DD996 and DD998, were crushed and split in two-halves by Zijin laboratories – one half was kept by Zijin laboratories for testing, while the other half was shipped to XPS in Canada.

The scope of work for both laboratories included:

- Sample head analysis in triplicate,
- Grind calibration curves, and
- Duplicate tests on DD996, DD998, and flotation composite 1 using the IFS4a flow sheet as developed during the Kamoa testwork programs. (See Figure 13.11).

Due to different flotation mechanisms in use at Zijin laboratories, the following adjustments were made in order to compare results directly to the XPS performance:

- Impeller speed of flotation mechanisms were increased to 1700rpm,
- Air addition method was changed from forced to self-induced,
- Scavenger recleaner stage reagent addition were moved to the scavenger cleaner feed, and
- Regrind media and mill speed were adjusted to suit the mill type.

High-grade concentrate products were produced by applying the IFS4a flow sheet with self-induced air addition (IFS4b), as summarised in Table 13.21 below.

Table 13.21 Flotation Composite 1 Flotation Performance on IFS4b Flow Sheet

Sample	Mass pull %	Recovery % Cu	Final Concentrate Grades (%)					
			Cu	SiO_2	S	Fe	Al_2O_3	As
Flotation composite 1	6.6	85.7	52.8	14.3	15.3	4.4	3.5	<0.01
DD996	7.0	87.8	53.3	15.3	14.5	3.7	3.8	<0.01
DD998	6.3	84.0	50.8	17.5	14.0	5.8	4.4	<0.01

These results achieved by Zijin laboratories indicated that the Kakula material tested were similar to the Kansoko Sud and Kansoko Centrale material, and that material from these deposits could be processed in a common concentrator.

Following the successful testing of the flotation composite 1 sample, new samples, DD1005 and DD1007 (flotation composite 2), were sent to Zijin laboratories in September 2016. The aim of this was to verify metallurgical characteristics of higher-grade samples and to reconfirm if the Kakula material was compatible with the IFS4a and IFS4b flow sheet.

The scope of work included rougher kinetic testing, verification/baseline flotation test on IFS4a and two optimisation tests. Refer to Table 13.22 for a summary of the results obtained.

Table 13.22 Flotation Composite 2 Flotation Performance by Zijin Laboratories

Flow Sheet	Mass pull %	Recovery % Cu	Final Concentrate Grades (%)					
			Cu	SiO_2	S	Fe	Al_2O_3	As
IFS4b	12.3	85.0	55.6	13.7	14.2	3.8	3.9	0.01
Optimised flow sheet 1	12.4	86.2	56.1	11.4	15.5	3.7	3.2	<0.01
Optimised flow sheet 2	11.9	87.9	60.5	15.4	14.2	4.1	3.9	<0.01

The changes made from IFS4b to the optimised flow sheet 2 included the following:

- Slightly finer rougher feed grind (80% passing 51 µm), and
- Extended scavenger floatation time from 40 min to 50 min.

Further testing was conducted in September 2016, by XPS, on samples DD1012 and DD1036 (flotation composite 3). As with the flotation composite sample 2, the aim of this was to verify metallurgical characteristics of higher-grade samples and to reconfirm if the Kakula material was compatible with the IFS4a flow sheet.

The only change made to the Kamoa IFS4a flow sheet was to change the air addition method from forced air to self-induced, as well as the adjustments of collector addition to cater for the increase in Cu grade in the sample. The resulting flow sheet was termed IFS4c. Refer to Table 13.23 for a summary of the results obtained.

Table 13.23 Flotation Composite 3 Flotation Performance by XPS

Test Reference	Mass pull %	Recovery % Cu	Final Concentrate Grades (%)					
			Cu	SiO_2	S	Fe	Al_2O_3	As
IFS4c FT001	12.5	87.8	56.0	14.4	13.8	4.2	4.1	–
IFS4c FT003	12.4	87.5	56.1	13.3	14.8	4.0	3.4	–

These results once again proved that the Kakula material and Kansoko material could be processed in a common concentrator.

13.3.5 Kakula PFS Flotation Flow Sheet Development Testwork

Kamoa Copper SA contracted XPS, in 2017–2018, to conduct flotation flow sheet development work on the Kakula deposit as part of the Kakula PFS. The aim of this campaign was to further optimise the flow sheet following the successful results obtained during the testing of the flotation composite samples 1, 2, and 3. Ten drill core samples (DD1017TW1, DD1020TW1, DD1029TW1, DD1032TW1, DD1043TW1, DD1065TW1, DD1075TW1, DD1081TW1, DD1112TW1, and DD997TW1) were composited to form the Kakula PFS development master composite, with a resultant grade of 6.13% Cu.

The scope of work included the baselining of the final grind target against the Kamoa Phase 6 IFS4c flow sheet (IFS4a flow sheet with self-induced air flow and reagents adjusted for higher head grade), assessment of primary grind, and optimisation of pulp densities, reagents and additions, regrind circuit, and low entrainment cleaning.

13.3.5.1 Baselining Against Kamoa Phase 6 IFS4c

Two tests were conducted during which the IFS4c parameters were applied, in order to generate a baseline for the Kakula PFS master composite sample. The two baseline tests achieved similar results, producing a final product of 52.2% Cu while recovering 86.3% Cu. The SiO_2 grade in the final product was approximately 16%.

The Kakula PFS composite sample did not perform as well as the Kakula FC3 sample which achieved a Cu recovery of 87.5% at a final product grade of 56.1% Cu. The variance in performance can be attributed to the fact that the Kakula FC3 sample had a higher head grade (8.1% Cu) compared to the Kakula PFS sample (6.1% Cu). Changes in the mineralogy and grain sizes also had an effect (see Section 13.3.2).

13.3.5.2 Kakula Flow Sheet Development and Optimisaion

A number of tests were conducted to test the following parameters on the Kakula flotation flow sheet:

- Effect of changing the mainstream grind from 80% passing 38 µm to 80% passing 150 µm;
- Effect of self-aspirated aearation and forced aeration methods for rougher and cleaner circuits;
- Effect of increasing collector addition in the rougher and scavenger circuit, as well as phased dosing of collector;
- Optimisation of high-grade cleaner circuit kinetics by varying collector dosages and flotation residence times;
- Optimisation of scavenger cleaner circuit kinetics by varying collector dosages, phased collector additions, and flotation residence times;
- Optimisation of the rougher circuit by changing rougher pulp density from 25% to 34%;
- Reducing regrind costs by moving the regind step from the scavenger cleaner feed to the scavenger recleaner feed; and
- Optimisation of the final product grades by using low entrainment cleaning.

The following was noted:

Increasing the rougher pulp density to 35% did not impact on Cu recovery but did result in high silica recovery to the high-grade circuit. This can be managed by further cleaning.

Modifications to the cleaner circuit did not result in any significant changes in overall recovery and only ended up shifting the circuit performance up or down the grade-recovery curve.

Re-positioning of the regrind step, from the scavenger cleaner feed to the scavenger recleaner feed, reduces the mass reporting to the regrind circuit from 30% to 12% of the fresh feed. A small increase in collector addition, to the scavenger recleaner stage as well as an increase in scavenger recleaner residence time from 10 min to 18 min was needed to improve recleaner recovery kinetics. Shifting of Cu units from the high-grade circuit to the scavenger cleaner circuit did not improve scavenger cleaner unit recoveries.

Low entrainment cleaning tests was conducted to determine if the concentrate grades could be increased by reducing the amount of gangue carried over to the concentrate by means of entrainment. Better selectivity of Cu over Silica was achieved in the concentrate.

13.3.5.3 Kakula 2019 PFS Flow Sheet

Refer to Figure 13.30 for an illustration of the final Kakula 2019 PFS flotation flow sheet, and Table 13.24 for a summary of the flow sheet conditions.

This flow sheet achieved a final recovery of 85.6% Cu, while producing a concentrate product of 57.3% Cu and 12.6% SiO_2. This recovery is similar to the recovery achieved in the baseline tests, however, an improvement on the Cu and SiO_2 grades were made.

Figure 13.30 Kakula 2019 PFS Flow Sheet (Kamoa Copper SA, 2017)

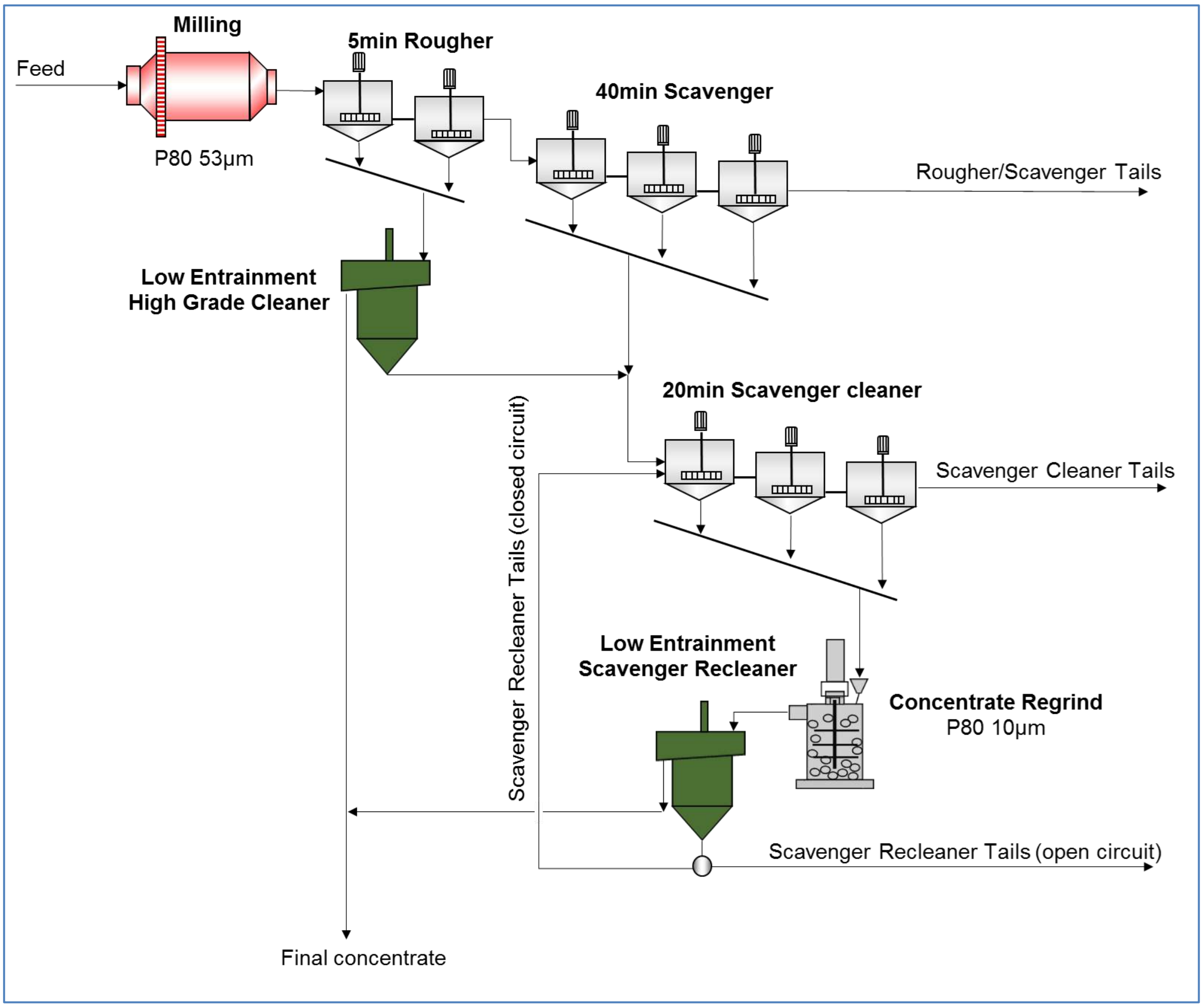


Table 13.24 Kakula 2019 PFS Flotation Parameters Summary

% Solids Grind	60								
Grind Target:	80% -53 µm								
Grind Media:	29.8 kg 440 SS (1" & ¾") rods								
Grind Time:	30:41								
Stage	**Cell Size % Solids**		**Est. Wt%**	**SIBX**	**3477**	**SF 22**	**Gas**	**RPM**	**Cum. Time**
Grind				179.0	32.0				
Ro Conc 1						76.0	5	1300	2
Ro Conc 2			~10-12%	26.0	5.0	9.0	5	1300	5
Ro Conc 3	4.5L	34%		17.0	3.0	16.0	9	1600	13
Ro Conc 4				17.0	3.0	16.0	9	1600	23
Ro Conc 5			~25%	17.0	3.0	16.0	10	1600	40
Rougher Conc 1-2 to High-grade Circuit									
High-grade Clnr 1	2.5L	10%	6%	17.0	3.0	10.0	4	1000	10
High-grade ReClnr 1	2.5L		5%				3	1000	9
High-grade ReClnr 2	2.5L						2	1000	7
Scav Clnr 1				0.0	0.0	10.0	5	1400	4
Scav Clnr 2				0.0	0.0		7	1600	13
Scav Clnr 3	4.5L	12-13%	30%	0.0	0.0	10.0	9	1600	17
Scav Clnr 4				12.4	2.2	10.0	9	1600	20
Regrind Combined Scav Cleaner Conc 1-4				12.4	2.2		Target P_{80} 10 µm		
Scav Rclner 1-1						10.0	3	1000	3
Scav Rclner 1-2	2.5L	8%	11%			10.0	5	1000	8
Scav Rclner 1-3				12.4	2.2	10.0	6	1000	28
Scav Rclner 2	2.5L	3%	5%				6	1000	20
Scav Rclner 3	2.5L	3%	4%				5-7	1000	15
Total				**310.2**	**55.6**	**203.0**			

13.3.5.4 Flotation Products Mineralogy

Mineralogy was conducted on a single rougher tailings sample, to determine the major cause of Cu losses to this stream. The Cu deportment indicated 86% of Cu to sulphides, of which the majority was chalcocite. This indicated that poor liberation was at fault for these Cu losses, rather than mode of occurrence. The average grain size of the Cu minerals in the rougher tailings was 3 µm – 5 µm. Almost 92% of the Cu sulphide minerals in the rougher tailings was locked – none of the Cu sulphide minerals in the rougher tailings were noted as being contained within the free or liberated classes.

Mineralogy was also conducted on a bulk concentrate sample, produced during the development campaign. Refer to Figure 13.31 for an illustration of the bulk modal and gangue liberation information. The modal analysis showed that 81.8% of all minerals occurred as Cu sulphides of which 86% of the Cu as sulphides occurred as chalcocite, 11% as bornite and 1.5% as chalcopyrite. The main gangue minerals in the concentrate were feldspar, quartz and Fe (Ti) oxides. Approximately 35% of the gangue that reported to the concentrate was in the free and liberated liberation classes.

Refer to Table 13.25 for the results of the full chemical analysis conducted on the Kakula PFS composite sample final concentrate product.

Figure 13.31 Kakula PFS Concentrate Modal Analysis and Gangue Liberation

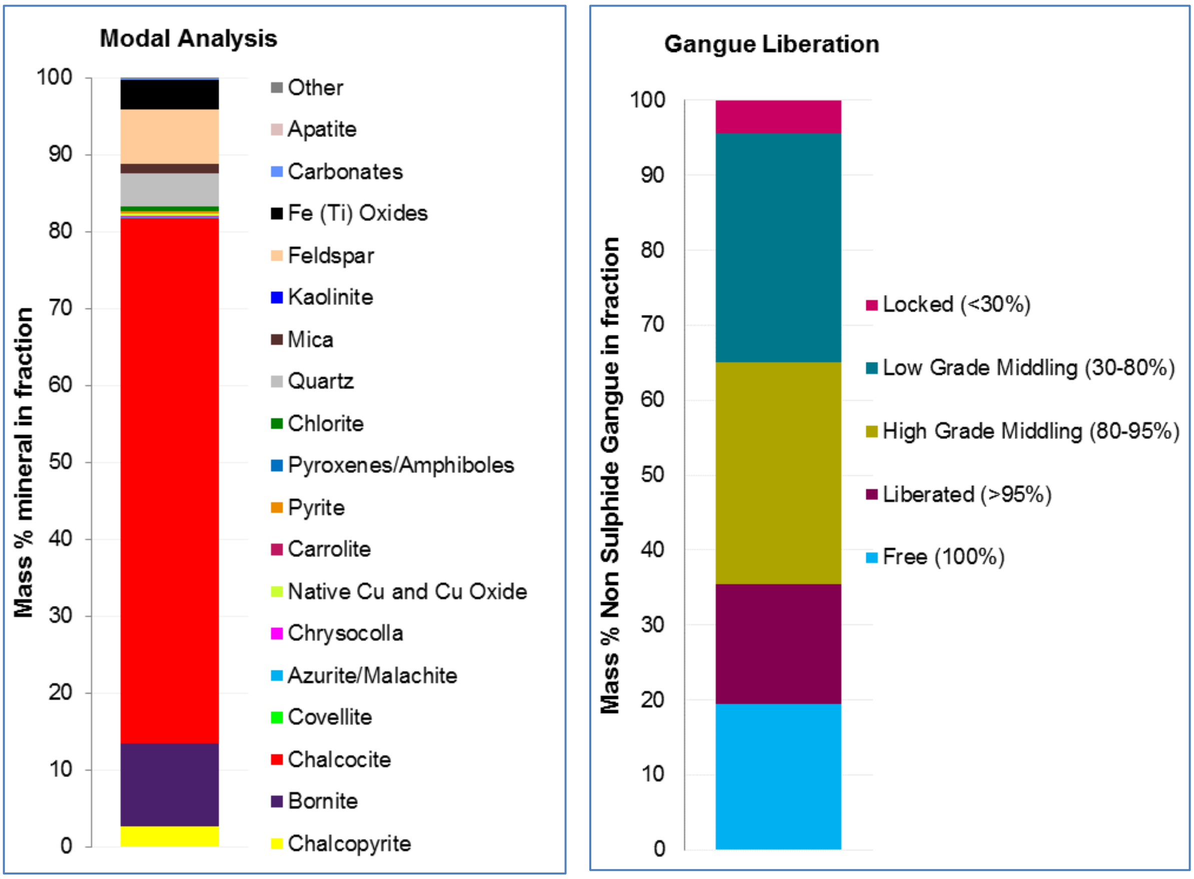


Table 13.25 Kakula PFS Concentrate Analysis

Element	Units	High-grade Concentrate	Recleaner Concentrate	Combined Concentrate
Mass %	**%**	**4.69**	**4.20**	**8.89**
Cu	%	72.16	40.71	57.32
Fe	%	2.52	8.04	5.13
S	%	18.70	12.52	15.79
As	%	0.01	0.01	0.01
SiO_2	%	3.11	23.18	12.58
MgO	%	0.30	1.58	0.91
Al_2O_3	%	2.05	5.69	3.77
CaO	%	0.51	0.60	0.56
As	ppm	< 5	12.00	5.66
B	ppm	< 10	50.00	23.60
Ba	ppm	25.00	116.00	67.95
Be	ppm	< 3	< 3	0.00
Bi	ppm	33.00	42.00	37.25
Cd	ppm	< 2	< 2	0.00
Ce	ppm	6.40	29.20	17.16
Co	ppm	41.90	135.00	85.84
Cr	ppm	< 30	580.00	273.72
Cs	ppm	1.00	2.80	1.85
Dy	ppm	1.20	4.20	2.62
Er	ppm	0.70	2.30	1.46
Eu	ppm	0.10	0.60	0.34
Fe	%	2.23	7.86	4.89
Ga	ppm	0.90	5.00	2.83
Gd	ppm	0.90	3.10	1.94
Ge	ppm	< 0.70	< 0.70	0.00
Ho	ppm	0.20	0.80	0.48
Hf	ppm	< 10.00	< 10.00	< 10.00
In	ppm	< 0.20	0.30	0.14
K	%	0.30	1.40	0.82
La	ppm	2.90	13.60	7.95
Li	ppm	4.00	21.00	12.02
Mn	ppm	58.00	92.00	74.05
Mo	ppm	10.00	28.00	18.49

Element	Units	High-grade Concentrate	Recleaner Concentrate	Combined Concentrate
Nb	ppm	4.40	17.90	10.77
Nd	ppm	2.80	12.10	7.19
Ni	ppm	< 10.00	100.00	47.19
Pb	ppm	27.00	73.10	48.76
Pr	ppm	0.80	3.20	1.93
Rb	ppm	5.10	46.70	24.73
Sb	ppm	< 2.00	< 2.00	0.00
Se	ppm	< 0.8	< 0.8	0.00
Si	%	1.23	8.46	4.64
Sm	ppm	0.60	2.40	1.45
Sn	ppm	5.90	2.80	4.44
Sr	ppm	6.00	18.00	11.66
Ta	ppm	0.20	0.90	0.53
Tb	ppm	0.20	0.70	0.44
Te	ppm	< 6.00	< 6.00	< 6.00
Th	ppm	1.50	5.10	3.20
Ti	%	0.09	0.40	0.24
Tl	ppm	0.40	0.60	0.49
Tm	ppm	0.10	0.30	0.19
U	ppm	0.80	2.60	1.65
V	ppm	12.00	47.00	28.52
W	ppm	< 0.7	2.00	0.94
Y	ppm	7.50	24.40	15.48
Yb	ppm	0.60	1.90	1.21
Zn	ppm	100.00	360.00	222.70

13.3.6 Preliminary Flotation Variability Campaign

13.3.6.1 Sample Characterisation

Ten kilograms of each sample was set aside for variability testing, prior to preparation of the Kakula PFS flotation development master composite. Head grade analysis for each sample was conducted in triplicate, and is summarised in Table 13.26.

Table 13.26 Kakula Preliminary Flotation Variability Samples Head Grade Analysis

Sample	Cu %	S %	SiO_2 %	Fe %	Al_2O_3 %	CaO %	MgO %	As %
DKMC_DD1017TW1	8.03	2.13	52.60	4.50	13.17	1.03	3.96	0.001
DKMC_DD1020W1	9.24	2.22	55.33	5.02	14.00	1.04	3.56	0.002
DKMC_DD1029W1	2.64	0.65	56.10	4.84	13.50	2.27	4.20	0.001
DKMC_DD1032W1	4.96	1.20	55.53	4.87	12.77	1.86	4.75	0.001
DKMC_DD1043W1	5.97	1.48	56.73	4.87	13.90	1.84	4.16	0.003
DKMC_DD1065W1	5.58	2.20	55.57	5.35	14.47	1.25	3.69	0.001
DKMC_DD1075W1	5.34	1.04	56.17	4.62	13.73	0.67	3.77	0.001
DKMC_DD1081W1	5.29	1.25	54.43	5.32	13.37	1.89	4.26	0.002
DKMC_DD1112W1	5.17	1.47	54.50	5.79	13.07	1.55	4.13	0.001
DKMC_DD997TW1	6.66	1.61	52.20	4.87	13.30	1.78	3.95	0.001

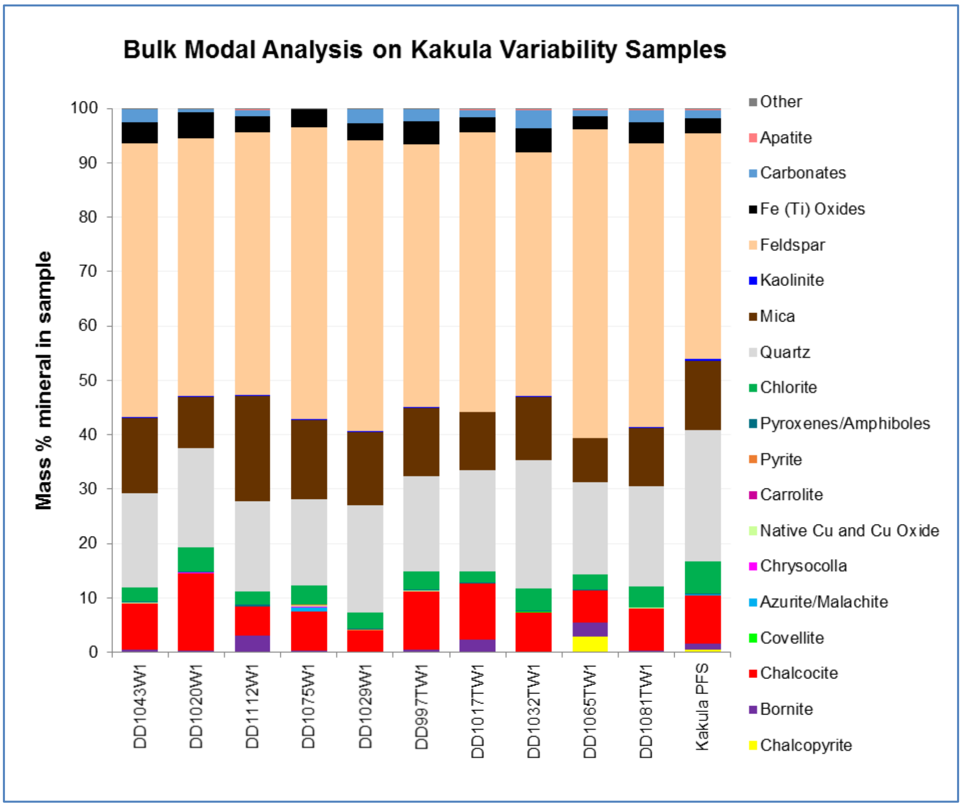
The following was noted from the head grade analysis:

- The samples tested varied from 2.6% Cu to 9.2% Cu, with sulphur grades generally increasing with increasing Cu grades.
- Fe, MgO, and Al_2O_3 values were relatively constant over the range of samples, averaging 5.0%, 4.0%, and 13.5% respectively.
- The highest As value measure was 0.003% for sample DD1043W1, with the majority of the samples reported as below the instrument detection limit of 0.001%.
- CaO, and SiO_2 values were variable.
- The Kakula samples were higher in Ca and Mg compared to the Kamoa Phase 6 material.

Bulk modal analysis on the minerals was conducted on each of the samples tested, of which the results are illustrated in Figure 13.32, while liberation data for each sample at a target grind of 80% passing 53 µm is presented in Figure 13.33.

In general, the Kakula material is significantly higher in feldspar compared to Kamoa Phase 6 material. XPS reported difficulty in filtration and settling of samples, due to the fine and ultrafine feldspar components. A varying carbonate content over the samples were noted. Chalcocite remains the main Cu minerals on all samples, however, the ratios of chalcocite, bornite, and chalcopyrite varied across all samples. Only sample DD1065W1 reported elevated levels of chalcopyrite. Sample DD1075W1 was the only sample with higher levels of poor-floating Azurite detected, and showed the lowest entitlement of sulphide Cu at 86%.

Figure 13.32 Kakula Preliminary Flotation Variability Samples Mineralogy Summary



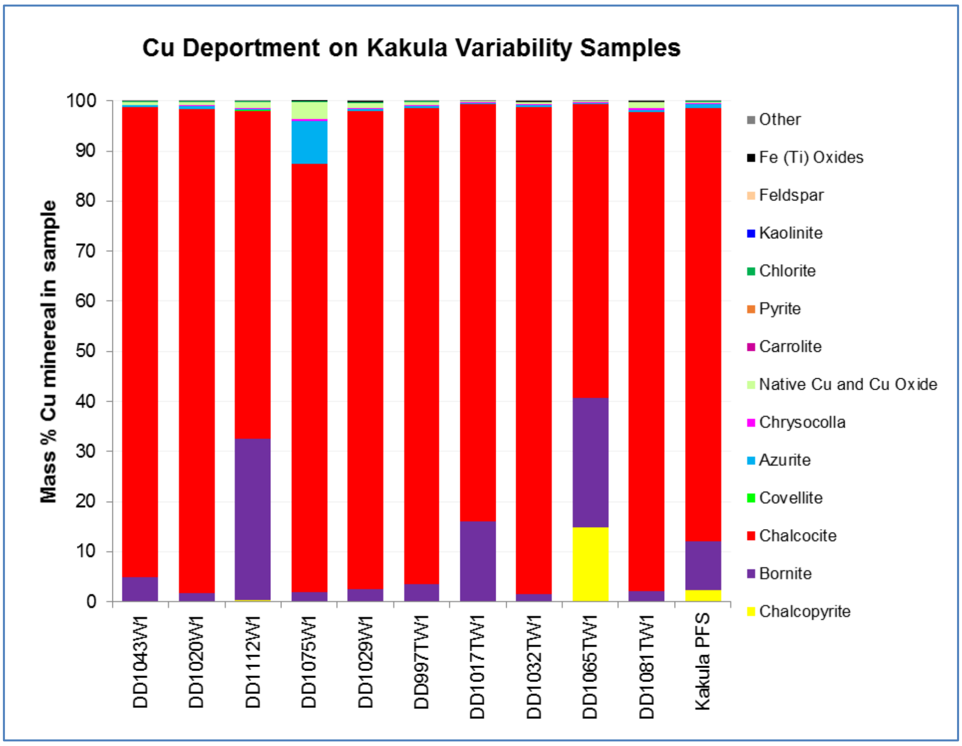
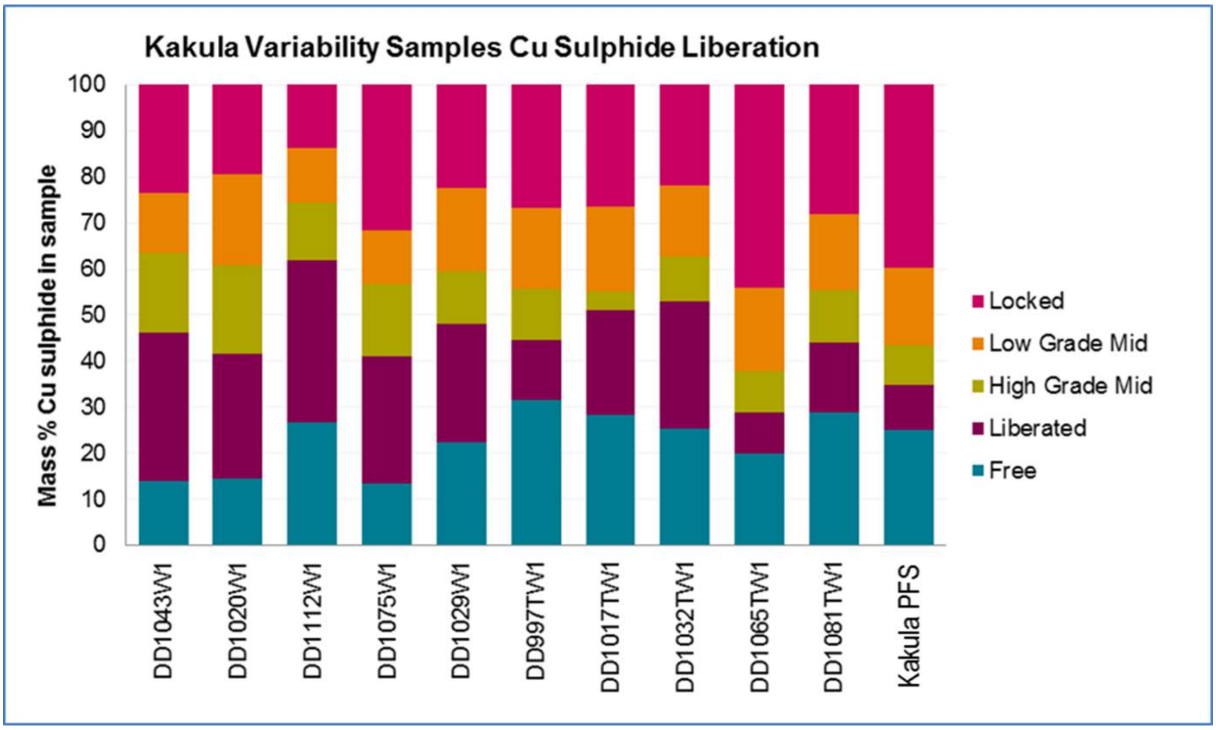


Figure 13.33 Kakula Preliminary Flotation Variability Samples Liberation Data



The Cu sulphide minerals occurring in the free and liberated classes, in the samples, were low at approximately 50%. This is consistent with expectations due to the fine grained nature of the sulphides. Liberation at a particle size of 80% -220 µm varied from 30% to 60% of the mass of Cu sulphides occurring as free or liberated grains, while the mass proportion of the locked Cu sulphides varied from 15% to 45%. The average Cu sulphide grain sizes varied significantly from 8 µm to 20 µm across the samples tested.

13.3.6.2 Flotation Results Summary

Grind calibration curves were completed for each of the individual samples, after which each sample was tested on the Kakula 2019 PFS flow sheet. Collector dosages were adjusted to a maximum of 50 g/t total collector for each percentage of Cu in the head – to allow for changes in head grade across the samples. The ratio of SIBX: Areo3477 was maintained at 85%:15%. Refer to Table 13.27 for a summary of the final concentrates produced in each of the tests.

Table 13.27 Kakula Preliminary Flotation Variability Results

Sample	Mass pull %	Head Cu %	Cu Recovery %	Final Concentrate Product		
				Cu %	SiO_2 %	Fe %
DD1043W1	6.5	6.0	79.0	70.0	4.5	1.9
DD1020W1	11.2	9.2	84.3	67.5	5.1	3.2
DD1112W1	7.1	5.2	90.8	65.9	4.2	5.0
DD1075W1 (Average)	4.8	5.3	64.7	70.6	5.4	2.2
DD1029W1	3.0	2.6	86.9	73.3	2.8	1.5
DD997TW1	7.6	6.7	81.9	73.1	4.5	2.1
DD1017TW1	10.7	8.0	87.9	70.3	4.1	2.6
DD1032W1 (2)	5.7	5.0	82.1	68.4	6.4	2.9
DD1065W1 (Average)	9.5	5.6	81.0	47.2	13.8	10.5
DD1081W1	6.8	5.3	86.3	64.5	8.5	3.1

The data indicated that the chalcocite rich samples produced similar results with Cu recoveries over 80% and SiO_2 grades below 10%. The sample rich in chalcopyrite (DD1065W1) only achieved an average grade of 47% Cu product at 81% Cu recovery, and high SiO_2 at 13.8%. Sample DD1075W1 was elevated in non-sulphide Cu and achieved the lowest Cu recovery at 64.7%.

Overall, the samples tested across the Kakula deposit performed relatively consistently on the Kakula flow sheet. The Cu mineralogy is variable and ratios between chalcocite, bornite, chalcopyrite and non-sulphide Cu are not consistent across the Kakula orebody. This variability in mineralogy resulted in changes of final concentrate grade and froth characteristics. Concentrate grades in excess of 64% were achieved on all samples except for DD1065W1.

No correlation was noted between Cu feed grade and final Cu recovery, but did impact on the final mass pull to the product. It was observed that higher proportions of Cu was recovered in the scavenger cleaner circuit as the head grade increased. The lower feed grade samples presented poorer frothing characteristics, while the higher-grade samples benefited from longer retention times in the scavenger cleaner circuit.

No correlation was noted between the Cu feed grade and final Cu recovery, however the Cu feed grade did impact on the expected mass pull to the final product and a correlation could be established between the mass pull and Cu upgrade ratio (UGR) to final product.

13.3.7 Other Testwork

13.3.7.1 HPGR Testwork

In March 2018, as part of the Kakula PFS phase, ThyssenKrupp conducted HPGR (High Pressure Grinding Roll) scoping testwork on Kakula material, at their testing facilities in Chloorkop, South Africa. The aim of this testwork campaign was to determine if the Kakula material was viable for processing via HPGR technology. The key parameters obtained from this campaign was:

- Specific throughput rate, m-dot in ts/h.m$_3$,
- Specific pressure force required in N/mm^2,
- Specific energy consumption in kWh/t, and
- Power requirement (kW) for a certain throughput (t/h) and roll size (m).

The testwork was conducted on a laboratory scale HPGR (LABWAL) and a wear test HPGR machine (ATWAL). These tests were conducted on roughly 135 kg of minus 12 mm sample remnants, from the Mintek PFS comminution testwork campaign.

Four single pass LABWAL tests were conducted at three different pressure settings, and a single run testing a high feed moisture content in the sample. Due to limited sample available at the time, only a single ATWAL test was conducted.

The following was noted on the Kakula PFS sample tested:

- The sample tested showed a low tendency to abrasiveness.
- The specific throughput of the sample averaged 280 ts/h.m^3 at 3.0% feed moisture. This is slightly higher compared to similar ores tested.
- In terms of product fineness, the sample tested fairly moderated compared to similar ore types tested.
- The anticipated specific grinding force required for industrial operations with studded rolls would be 1.5 – 3.0 N/mm^2.
- An increase of feed moisture in the sample from 3.0% to 5.0% resulted in a 5.0% reduction in throughput rate.

In general, the Kakula PFS material was noted as being well suited for treatment in an HPGR. No process guarantees could be given by the vendor, based on scoping testwork alone, and pilot scale testwork was required.

Following the successful scoping testwork, in October 2018, ThyssenKrupp was contracted to conduct pilot plant scale HPGR testwork, on the Kakula material.

The aim of the pilot plant campaign was to confirm the findings from the scoping study to a level that an industrial unit could be designed and scaled up and process guarantees be given. Key parameters, similar to the scoping study, i.e. specific throughput, pressing force, energy consumption and power requirements was obtained. The testwork was conducted using a semi-pilot scale HPGR (SMALLWAL) and a wear test HPGR machine (ATWAL).

The pilot scale testing was conducted using Kakula diamictite and sandstone material. The following conclusions were made following the pilot testwork:

- The ATWAL abrasiveness test confirmed that the Kakula material has a low tendency to abrasiveness.
- The average SMALLWALL specific throughput of the two samples was 285 ts/h.m^3 at 3.0% feed moisture and a specific grinding force of 2.5 N/mm^2. This is slightly higher compared to similar ores tested.
- An increase in specific grinding force leads to a decrease in throughput – increasing the specific grinding force to 3.5 N/mm^2 resulted in a 9% decrease in throughput to 273 ts/h.m^3.
- Higher grinding forces resulted in higher power draw – the specific energy requirement increased from 1.8 kWh/t to 2.25 kWh/t when increasing the specific grinding force from 2.5 N/mm^2 to 3.5 N/mm^2.
- The effect of increased moisture content was worse on the diamictite sample – an increase in moisture from 3.0% to 5.0% resulted in a throughput reduction from 287 ts/h.m^3 to 267 ts/h.m^3, compared to a drop from 287 ts/h.m^3 to 276 ts/h.m^3 for the sandstone sample. The effect of increased moisture content did not have any impact on the fineness of the products produced.
- The effect of pre-screening the fines fraction from the HPGR feed resulted in lower specific throughputs – 263 ts/h.m^3 for the diamictite sample and 244 ts/h.m^3 for the sandstone sample.
- The fineness of the products produced were similar for the two samples tested.

BBWi and grindmill testing was conducted by Mintek in 2018 on product material from the HPGR pilot plant campaign. The HPGR crushed material reported a lower BBWi compared to conventionally crushed material, as per Table 13.28.

Table 13.28 Kakula PFS HPGR Product BBWi Data at 75 µm Screen

Sample ID	BBWi - HPGR Crushed kWh/t	BBWi – Conventionally Crushed kWh/t
Diamictite	15.8	17.2
Sandstone	16.9	17.8

13.3.7.2 Bulk Material Flow Testwork

Bulk material flow testing was conducted by GreenTechnical, during April 2018, to facilitate with material handling designs. Product sample from the HPGR scoping test was used for this campaign. The scope of work included a number of flow property tests: Jenike shear cell, wall friction, compressibility, moisture content, and chute friction angle test.

13.3.7.3 Concentrate Thickening Testwork

During July 2018, the Outotec Testing Facility in Sudbury, Canada conducted settling testwork on a Kakula PFS final concentrate composite sample, as prepared as part of the flotation flow sheet development campaign by XPS. The aim of the testing was to determine the optimum thickener design and operating parameters. The testing included material characterisation, flocculant selection, and batch dynamic thickening. The material characteristics, as determined by Outotec, are presented in Table 13.29.

Table 13.29 Kakula PFS Flotation Concentrate Characteristics (Outotec)

Parameter	Value
Slurry pH	8.1
Slurry P_{50}	19.0 µm
Slurry P_{80}	47.8 µm
Specific gravity	4.85

The bench-top dynamic thickening tests indicated that an underflow solids concentration of 72.5% could be obtained from a solids flux rate of 0.25 t/m^2.h. The overflow clarity achieved, with a flocculant dosage of 30 g/t, was 216 mg/l solids to overflow, while the overflow clarity improved to 137 mg/l solids to the overflow with a flocculant dosage of 40 g/t. A yield strength of 99 Pa was measured at a solids underflow concentration of 72.5%.

13.3.7.4 Concentrate Filtration Testwork

Following the thickening testwork, Outotec conducted further testwork on the Kakula PFS concentrate sample to determine the suitability of the Larox® Pressure Filter (PF) and Fast Filter Press (FFP) technology for dewatering of the material. Bench scale testing was conducted to evaluate filter cloth selection, filter cake thickness, filtration rate, cake moister content, and filter cake handling characteristics. A summary of the findings are presented in Table 13.30.

Table 13.30 Kakula PFS Final Concentrate Filtration Testing Results Summary

Dewatering Technology	Test	Air Drying Time Minutes	Filtration Rate kg DS/m^2.h	Cake Moisture % w/w	Cake Thickness mm	Pressing Pressure Bar	Air Pressure Bar
PF Pressure Filter	1	3.0	840	5.8	52	16	9
PF Pressure Filter	2	1.0	1037	7.7	52	16	9
FFP Fast Filter Press	3	3.0	510	6.7	58	12	9

This testwork indicated that the Kakula PFS final concentrate product could be successfully dewatered to within the targeted moistures (8% – 10%), at high solid flux rates.

13.3.7.5 Tailings Thickening, Rheology and Filtration Testwork

During June 2018, SGS Canada conducted solid-liquid separation, rheology, and pressure filtration testwork on a Kakula PFS final tailings composite sample, as prepared as part of the flotation flow sheet development campaign by XPS. The aim of the testing was to determine the optimum thickener design and operating parameters. The testing included material characterisation, flocculant selection, static settling, and batch dynamic thickening. The material characteristics as determined by SGS are presented in Table 13.31, and compared to the Kamoa Phase 6 tailings sample.

Table 13.31 Kakula PFS Flotation Tailings Characteristics (SGS)

Parameter	Kakula PFS Tailings	Kamoa Phase 6 Tailings
Slurry pH	7.8	7.2
Slurry P_{80}	48 µm	41 µm
Specific gravity	2.87	2.77

Flocculant scoping tests indicated that the Kakula PFS sample required sequential dosing of BASF Magnafloc 380 followed by BASF Magnafloc 10 (Kamoa Phase 6 sample required single dosage of BASF Magnafloc 10). Refer to Table 13.32 for a summary of the preliminary static settling test results.

Table 13.32 Kakula PFS Static Settling Test Result Summary

Parameter	Units	Kakula PFS Tailings	Kamoa Phase 6 Tailings
Flocculant 1 type	–	BASF Magnafloc 380	BASF Magnafloc 10
Flocculant 1 dosage	g/t	45.0	35.0
Flocculant 2 type	–	BASF Magnafloc 10	N/A
Flocculant 2 dosage	g/t	25.0	N/A
Feed solids density	% w/w	5.0	10.0
Underflow solids density	% w/w	49.0	53.0
Critical solids density (CSD)	$m^2/(t/d)$	58.5	58.0
Thickener unit area	$m^2/(t/d)$	0.20	0.11
Initial settling rate	$m^3/m^2/d$	625	536
Overflow TSS (clarity)	mg/L	28 (hazy)	<10 (clear)

After the completion of the static test, dynamic settling tests were conducted to determine the effect of changing flocculant dosage with a constant thickening area, and the effect of changing thickening area while keeping the flocculant dosage constant. Refer to Table 13.33 for a summary of the effect of reducing flocculant dosage rates at a fixed unit area of 0.22 m^2/t/d.

Table 13.33 Effect of Flocculant Dosage on Overflow Clarity for Kakula PFS Tailings

Thickener Unit Area	Magnafloc 380	Magnafloc 10	Overflow Clarity
0.22 m^2/(t/d)	50	30	33 mg/L
0.22 m^2/(t/d)	45	25	50 mg/L
0.22 m^2/(t/d)	40	20	99 mg/L

Refer to Table 13.34 for a summary of the effect of reducing unit area at fixed flocculant dosage rates (45 g/t Magnafloc 380 followed by 25 g/t Magnafloc 10).

Table 13.34 Effect of Thickening Area on Settling Parameters at Constant Reagent Dosage

Thickener Unit Area m^2/(t/d)	Solids Loading t/m^2/h	Nett Rise Rate m^3/m^2/d	Underflow density % solids w/w	Overflow TSS mg/L	Residence τ h
0.22	0.19	80	59.0	50	2.27
0.20	0.21	88	58.8	70	2.07
0.18	0.23	98	57.5	122	1.86
0.16	0.26	110	56.8	145	1.65
0.14	0.30	126	55.0	181	1.45

The rheology test indicated that the Kakula PFS sample displayed a Bingham plastic response and had a critical solids density (CSD) of 58.5% solids (w/w) which corresponded to a yield stress of 42 Pa under un-sheared conditions, and 18 Pa under sheared conditions (compared to 27 Pa and 22 Pa respectively for the Kamoa Phase 6 tailings sample tested).

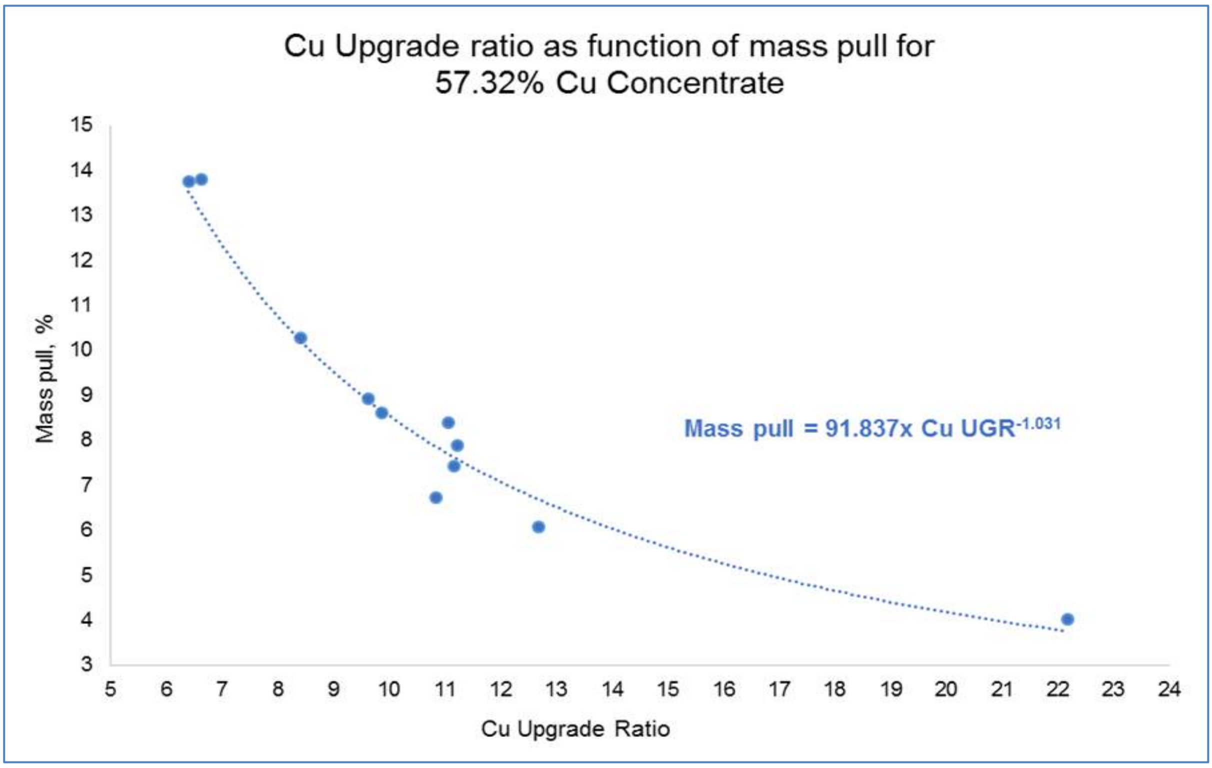
Pressure filtration tests were conducted using the flotation tailings thickener underflow material at a feed density of 59.0% solids (w/w). The tests were conducted at pressure levels between 6.9 bar and 9.9 bar. The test cake thickness ranged from 14 mm to 31 mm, while the resulting solids throughput ranged from 578 kgDS/m^2.h to 977 kgDS/m^2.h The residual cake moisture varied between 15.9% solids (w/w) to 18.6% solids (w/w).

13.3.8 Kakula 2019 PFS Recovery Estimate

The recovery estimate, for the Kakula 2019 PFS, is based on the test information generated by the Kakula PFS campaign and the Kakula preliminary variability tests. The recovery model targets a final product grade of 57.3% Cu, as per the Kakula PFS composite sample performance on the final Kakula flow sheet.

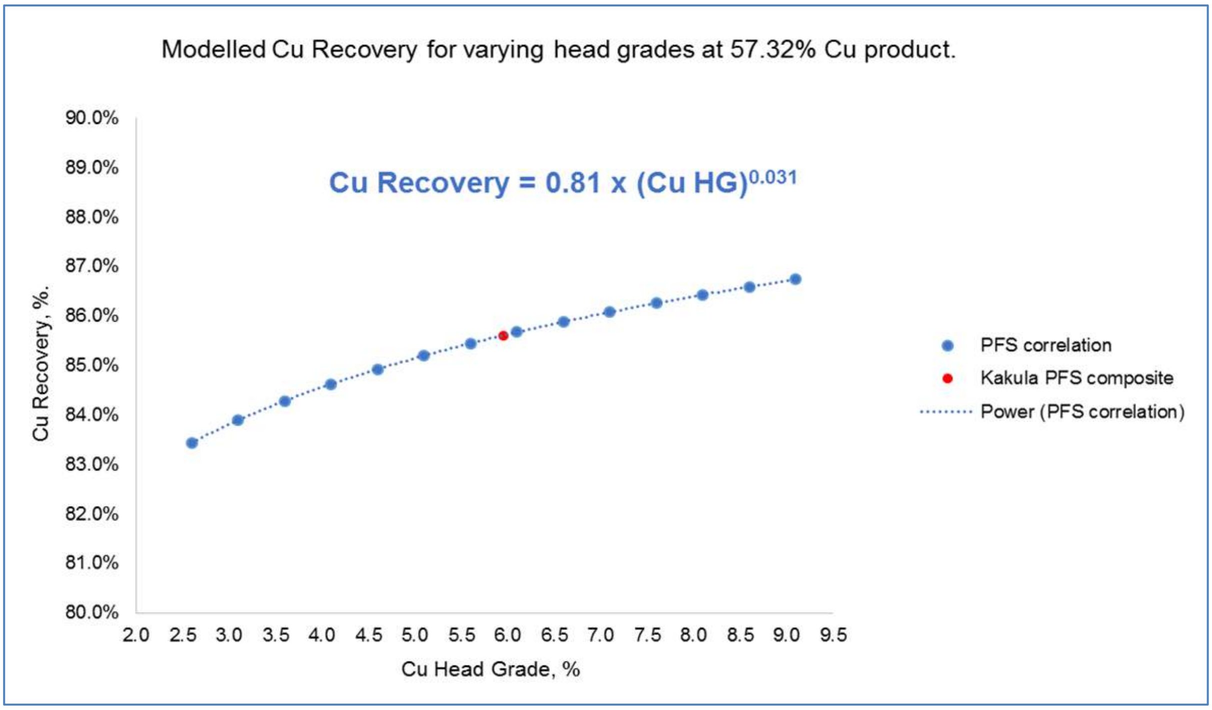
No correlation was noted between the Cu feed grade and final Cu recovery, however the Cu feed grade did impact on the expected mass pull to the final product and a correlation could be established between the mass pull and Cu upgrade ratio (UGR) to final product. This information was obtained from the individual Cu UGR vs mass pull curves. Targeted UGRs was calculated by dividing the targeted final product grade (57.3%) by the individual back calculated head grades from each of the tests, and the associated mass pulls noted. The data is presented in Figure 13.34.

Figure 13.34 Kakula Cu UGR vs Mass Pull Correlation



The resulting correlation from Figure 13.34 is used the calculate the expected mass pull for varying head grades, by determining the targeted Cu upgrade ratio based on a 57.3% final product. The associated Cu recovery is then calculated using the mass pull and concentrate grade. Refer to Figure 13.35 for the Kakula Cu recovery algorithm.

Figure 13.35 Kakula Cu Recovery as a Function of Cu Head Grade



13.4 Preliminary Testwork on Kakula-West Material

In 2018, XPS conducted mineralogy and flotation tests on a single Kakula West composite sample.

13.4.1 Kakula West Sample Details and Characterisation

A total of 12 samples, from four holes representative of the envisaged Kakula West mining area, were delivered to XPS towards the last quarter of 2018. The details of the various samples are presented in Table 13.35. This material was composited into a single sample for testing. Head analysis were conducted in triplicate, on the Kakula West composite sample, and is summarised in Table 13.36.

Table 13.35 Kakula West Drillhole Details

Drillhole ID	Depth From m	Depth to m	Sample Mass kg	Expected Cu Grade % Cu
DKMC_DD1152	456.6	458.8	5.2	2.01
DKMC_DD1177	568.6	570.2	5.1	5.72
DKMC_DD1180	491.9	494.2	5.8	2.40
DKMC_DD1336	522.0	525.0	11.1	3.17

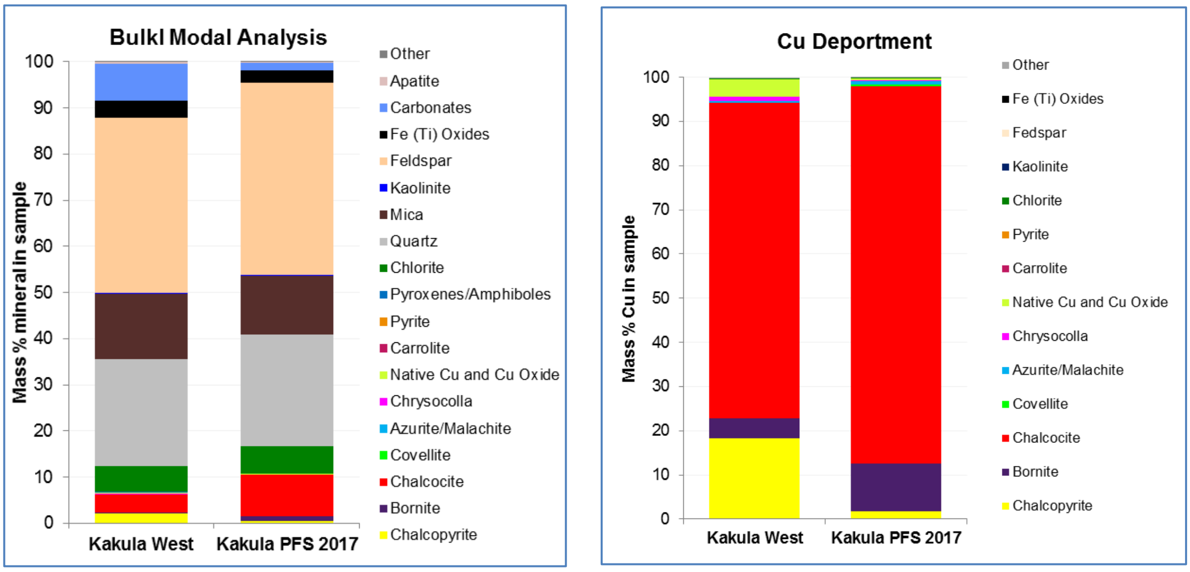
Table 13.36 Kakula West Flotation Composite Sample Head Analysis

Sample	Cu %	S %	SiO_2 %	Fe %	Al_2O_3 %	CaO %	MgO %	As %
Kakula West flotation composite sample	3.17	1.07	54.00	4.99	12.90	4.62	4.50	<0.01

A summary of the bulk modal analysis and Cu deportment study conducted on the Kakula West sample, at 80% passing 212 µm, is given in Figure 13.36. The Kakula West sample was lower in Cu grade compared to the Kakula PFS sample tested. The main Cu mineral in the Kakula West material was chalcocite, followed by chalcopyrite and smaller amounts of bornite. The Kakula West sample hosts higher levels of chalcopyrite than the Kakula PFS sample tested.

The Kakula West and Kakula PFS samples had similar levels of chlorites, quartz, and mica. The Kakula West sample showed slightly lower feldspar levels when compared to the Kakula PFS sample, but with higher carbonates. The average grain size of the Kakula West Cu sulphide minerals was noted as similar to the Kamoa Phase 6 sample - slightly coarser than the Kakula PFS sample tested.

Figure 13.36 Kakula West Sample Mineralogy



13.4.2 Flotation Performance on Kakula Flow Sheet

The Kakula West sample was tested in duplicate using the Kakula PFS flow sheet, and performed very well by achieving a final Cu recovery of 86.1% while producing a concentrate at 54% Cu and 8.6% SiO_2.

This indicates that the Kakula and Kakula West material can be treated in a common concentrator circuit.

13.5 Kamoa Phase 6 Sample Performance on Kakula Flow Sheet

XPS further tested the performance of the Kamoa Phase 6 signature plot composite sample (in duplicate) on the Kakula PFS flow sheet to compare performance of the sample to the IFS4a flow sheet.

The Kamoa Phase 6 signature plot composite sample achieved a final Cu recovery of 86.6% while producing a concentrate at 36.2% Cu and 13.0% SiO_2. This was poorer than the sample's performance on the IFS4a flow sheet which achieved 89.3% Cu recovery while producing a product at 36.7% Cu and 9.1% SiO_2. Changes in performance can be attributed to the following variances between the Kamoa and the Kakula flow sheets:

- Better performance on the Kakula rougher/scavenger and high-grade cleaning circuit due to changes in aeration methods and additional collector (Cu losses to rougher tailings reduced from 5.6% to 4.8%).
- Inferior performance in the Kakula scavenger circuit due to repositioning of the regrind stage (increase in scavenger cleaner and scavenger recleaner tailings Cu losses from 5.0% to 8.6%).

It did however indicate that the Kakula and Kamoa material can be treated in a common concentrator.

13.6 Comments on Section 13

In the opinion of the QP the metallurgical testwork conducted for the Kamoa and Kakula deposits is sufficient for PFS level process design. The comminution characteristics are well established and have consistency across the various testing phases and across the prospective mining areas.

Despite the variable mineralogy, the flotation characteristics are well understood and explainable in terms of the process mineralogy. The samples tested reasonably represent the material to be mined and processed according to the mine schedule.

The project mineralised zones do not contain deleterious elements often found in copper concentrates, such as arsenic and fluorine and Kakula is especially low in Arsenic. As a result, the flotation testwork has consistently generated concentrates that are free of penalty elements.

The pervasive presence of ultrafine copper sulphides in all Kamoa samples leads to strong recovery of silica through attachment with these sulphides. This, in turn, has led to high rougher mass pull rates and silica rejection challenges in final concentrate production, which is mitigated to a large degree by 10 µm regrinding of middling streams. The most recent testwork, at two independent laboratories, has consistently achieved silica levels in the range 14 to 15% SiO_2 and has provided confidence that this level of silica rejection, at a minimum, will be achievable in operations. Low entrainment cleaning in the Kakula circuit further facilitated in reducing silica leves in the final concentrate.

The power required to conduct ultrafine regrinding has been estimated for Kamoa deposit (using an IsaMill signature plot), and the results are reasonably consistent across the samples tested. The Kamoa testwork result has been used as an estimate of milling requirements for the Kakula PFS. No regrind testwork has been done to date on the Kakula material, however, this testwork is planned for Q1 2019.

The prediction of copper recovery from Kamoa hypogene samples is reasonable based on the testwork to date, while the prediction of copper recovery for the Kansoko surface-linked-oxidation supergene samples applicable is more complex and variable. A separate method of copper recovery prediction for Kamoa supergene mineralisation uses measured ASCu assay values to predict oxide copper recovery, where this is deemed necessary. It should be noted that the lack of surface supergene mineralisation, at Kakula, makes this matter irrelevant for that deposit.

The prediction of copper recovery for the Kakula material is based on variability testwork which compares well with the performance of the Kakula PFS sample used for flow sheet development.Compared to the Kamoa mineralised zones, the Kakula deposit has less variability in copper mineralisation, a low and consistent arsenic content and effectively equivalent comminution properties.

The Kamoa work performed to date is appropriate for the Kamoa 2017 PFS mine plan but will not be adequate should the mine plan change to incorporate significant mineralisation that has not yet been tested for either comminution or flotation response.

14 MINERAL RESOURCE ESTIMATES

14.1 Key Assumptions/ Basis of Estimate

The Kamoa and Kakula Mineral Resource models are two separate models within the Project area. The Kamoa Mineral Resource model was previously reported in the March 2018 Kamoa-Kakula Resource Update Technical Report and remains unchanged; the Kakula models are updated in this report. The qualified persons for the Kamoa-Kakula Mineral Resource estimates are Dr. Harry Parker, RM SME, and Mr Gordon Seibel, RM SME, employees of Amec Foster Wheeler. The Kamoa and Kakula Mineral Resource estimations were constructed by Mr. George Gilchrist, Pr. Sci Nat, Ivanhoe's Mineral Resources Manager.

The Kakula Mineral Resource model is divided by the prominent north–north-west-trending West Scarp Fault that has a significant vertical offset, into two sections. The Kakula West deposit is located west of the West Scarp Fault, and Kakula is located east of the West Scarp Fault. The Kakula West resource model was modelled using drillhole data provided up to 1 November 2018, and Kakula was modelled using drillhole data provided up to 26 January 2018.

Kamoa, Kakula and Kakula West resource models have now been updated from 2D to 3D estimation methods. The resource estimation methodology combines stratigraphic and mineralised units to constrain a full 3D block model with multiple vertical domains. Mineralised zones are defined using an approximate cut-off grade of 1% TCu (locally 0.5% TCu), and a minimum 3m vertical thickness was required for reporting the Mineral Resource to reflect the minimum underground mining height. At Kamoa, four mineralised domains were modelled in different stratigraphic positions. At Kakula, a single mineralised zone was modelled near, or just above, the Roan contact, which may be locally separated into two domains based on whether the host is a siltstone or diamictite unit.

To account for the undulations of the deposit and ensure that the grade profiles between drillholes align during estimation, drillhole composites and blocks were transformed vertically or "dilated" to a constant thickness that matched the maximum thickness of the domain. This method aligns the top, middle and bottom of the mineralised intervals horizontally for variography and grade estimation using ordinary kriging (OK). Both the variography and grade estimation were constrained vertically from a quarter to a third of the vertical dilated thickness. This preserves the vertical grade and mineralogical zonation to allow vertical optimisation during mine design. To adjust for local changes in the trend of the mineralisation laterally, geological controls were used to locally adjust the search orientations during estimation using a Datamine process known as dynamic anisotropy. This was used primarily at Kakula West.

Capping of assays was determined and performed by domain. At Kakula, higher-grade samples are well supported laterally and vertically, and hence no capping was applied to copper in the primary mineralised domain.

Amec Foster Wheeler considers the Mineral Resource models and Mineral Resource estimates derived from those models to be consistent with industry best practices (CIM, 2003) and to conform to the requirements of the CIM Definition Standards (2014).

14.2 Selective Mineralised Zones (SMZ)

14.2.1 Kamoa

The Mineral Resource estimate used 776 drillhole intercepts, which include drillholes within the mining lease, but excludes drillholes at Kakula and on the Kamoa and Makalu domes; these domes are areas where the favourable Ki1.1.1 stratigraphic unit is not present, or where the mineralisation has been completely leached. Included in the 776 drillholes are 16 twin holes (where the spacing between drillholes is <25 m) and six wedge holes. These drillholes were used in the estimation, and weightings assigned to these drillholes during estimation were scrutinised to ensure negative weights did not create estimation biases due to clustering that can result with close drillhole spacings.

Collar, survey, assay, stratigraphy and SG data were exported from the Ivanhoe acQuire database as a series of csv files, imported into Datamine Studio 3 mining software, and combined to form a desurveyed drillhole file. The entire Ki1.1.1 and Ki1.1.2 units were estimated, and if distinct mineralised zones within these units could be identified, they were modelled and estimated separately. The drillhole file was exported to Excel, and the SMZ selections in each hole were manually chosen.

In general, the SMZ selection was based on the 1.0% cut-off used in the 2D models. The basal contact of the SMZ is usually sharp and easily defined using the 1% TCu cut-off. In areas with gradational grade profiles (typically the top contact), a lower cut-off approaching 0.5% TCu was used, as a 1% TCu cut-off would locally truncate the gradational grade profile. Since the grade profile is often a function of the localised development of siltstone or sandstone layers, these layers were evaluated during the SMZ coding. The nature of the grade profile and the characteristics of surrounding drillholes are also a key consideration to ensure that the defined top and bottom contacts of the SMZ in any specific drillhole matched the same part of the grade profile as the top and bottom contacts of the SMZ defined in surrounding holes. This was to give consideration to the transformation and estimation methodology used, where the transformation is specifically designed to match the grade profile from one drillhole to the next to ensure the appropriate vertical position of the samples is correct for selection in the kriging search neighbourhood.

Two main mineralised zones were identified, the Upper SMZ and Lower SMZ. These SMZs occupy distinct positions vertically, and lateral extents are largely controlled by growth faults especially evident at Kansoko Sud (Figure 14.1). The Upper SMZ occurs across the majority of the Kamoa area north-east of the growth faults, and was locally subdomained in the Kansoko Sud area (Upper SMZ 2), where a bimodal grade distribution develops in response to changes in stratigraphy in a narrow zone (500 m wide) along the trace of the growth faults. The Lower SMZ occurs to the south-west.

The Upper SMZ is the most laterally continuous and best developed of the modelled minzones, and accounts for the vast majority of the Kamoa Mineral Resource. At Kansoko Sud, the intermediate siltstone (Ki1.1.1.2c) changes from a very narrow (or non-developed) siltstone to a much thicker zone of siltstone-sandstone-siltstone layers. The lower siltstone layer remains consistently mineralised (the Upper SMZ); the sandstone between the two siltstone layers is poorly mineralised, and the uppermost siltstone (Ki1.1.1.2.c) is moderately mineralised (Figure 14.2).

Figure 14.1 Plan View Showing Lateral Distribution of the Three SMZs

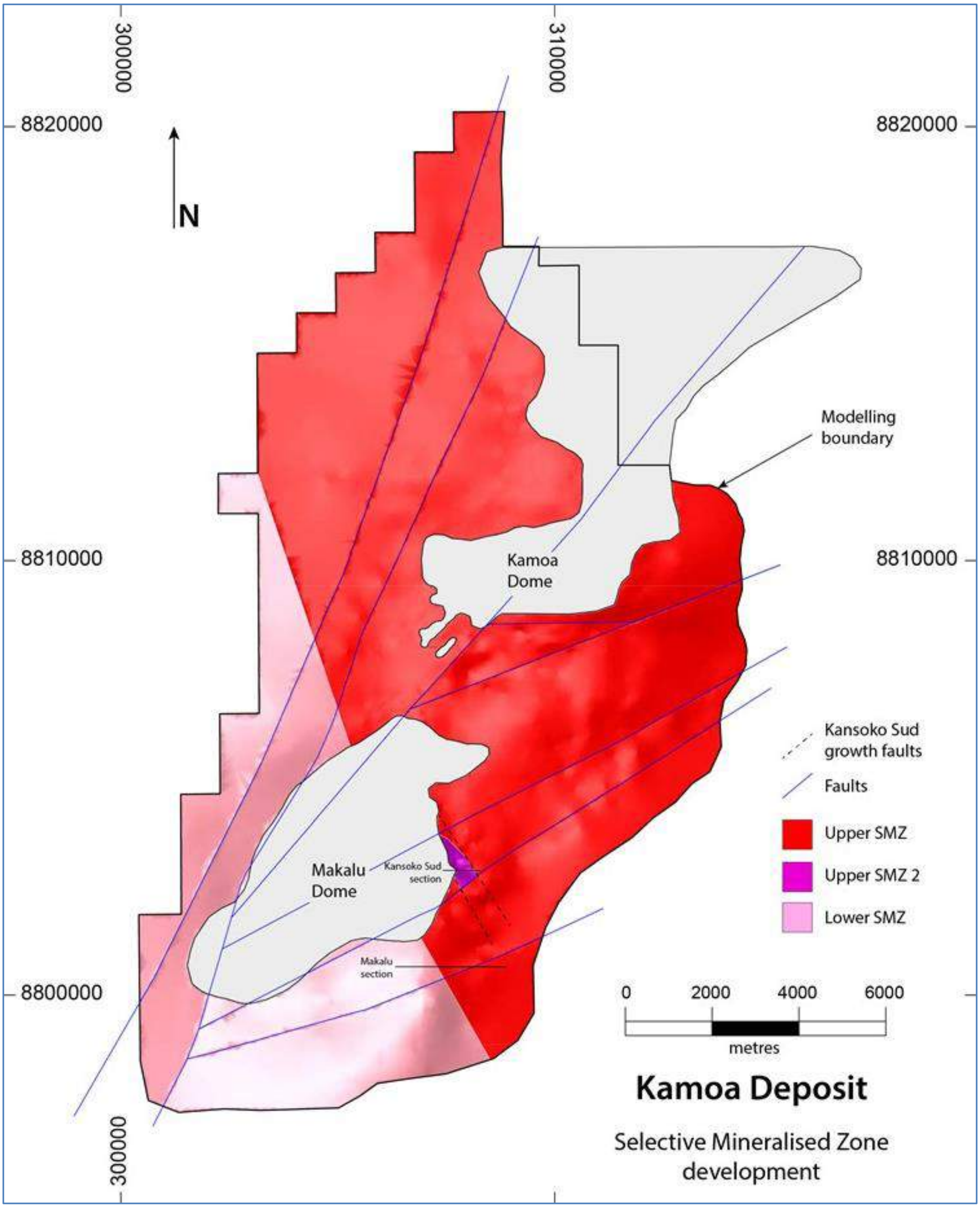


Figure provided by Ivanhoe, 2017.

Figure 14.2 Relationship of the Two Upper SMZ Zones Developed at Kansoko Sud

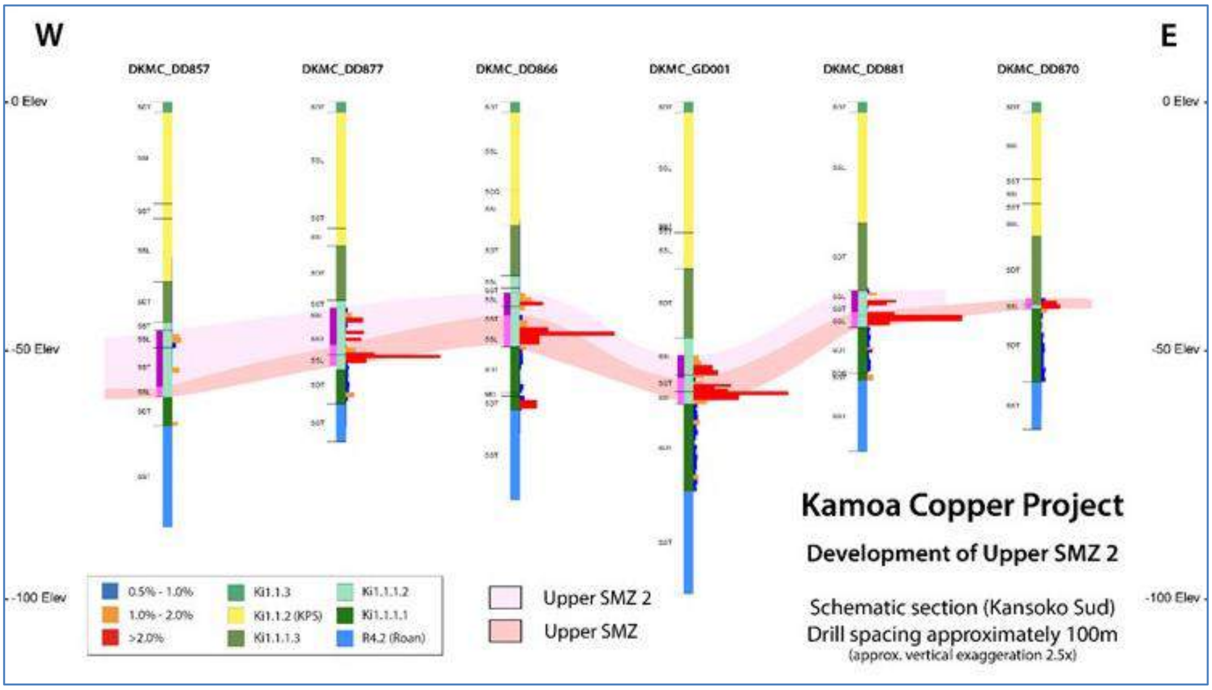


Figure provided by Ivanhoe, 2017. Section line (Kansoko Sud) shown in Figure 14.4.

To avoid smoothing of grades vertically during estimation, a separate SMZ was defined as a separate domain in these areas, incorporating grade in the sandstone and upper siltstone of the Ki1.1.1. 2c.

South-west of the growth faults at Kansoko Sud, the mineralisation in the Upper SMZ weakens, and a separate mineralised zone develops at the base of the Ki1.1.1.1, close to, or on, the R4.2 contact. This Lower SMZ is generally lower-grade than the Upper SMZ, but is recognised in both the Makalu area and in the Kamoa Ouest prospect area. A lack of drillholes in the southern portions of the Makalu prospect area make correlations with Kakula difficult; however, the mineralisation developed at Kakula occurs in the same stratigraphic position as the Lower SMZ. At Makalu, the lateral overlap between the Upper SMZ and Lower SMZ is approximately 800 m (Figure 14.3).

The assay file, with the SMZ selections flagged, was then imported into Datamine® Studio 3 mining software where it was combined with the collar and survey files. The SMZ selection fields were added to the de-surveyed drillhole files as a series of columns, with a value of '1' assigned where the samples were within a specific SMZ, and a default value of '0' for all other samples.

Figure 14.3 Relationship of the Upper SMZ and Lower SMZ Developed at Makalu

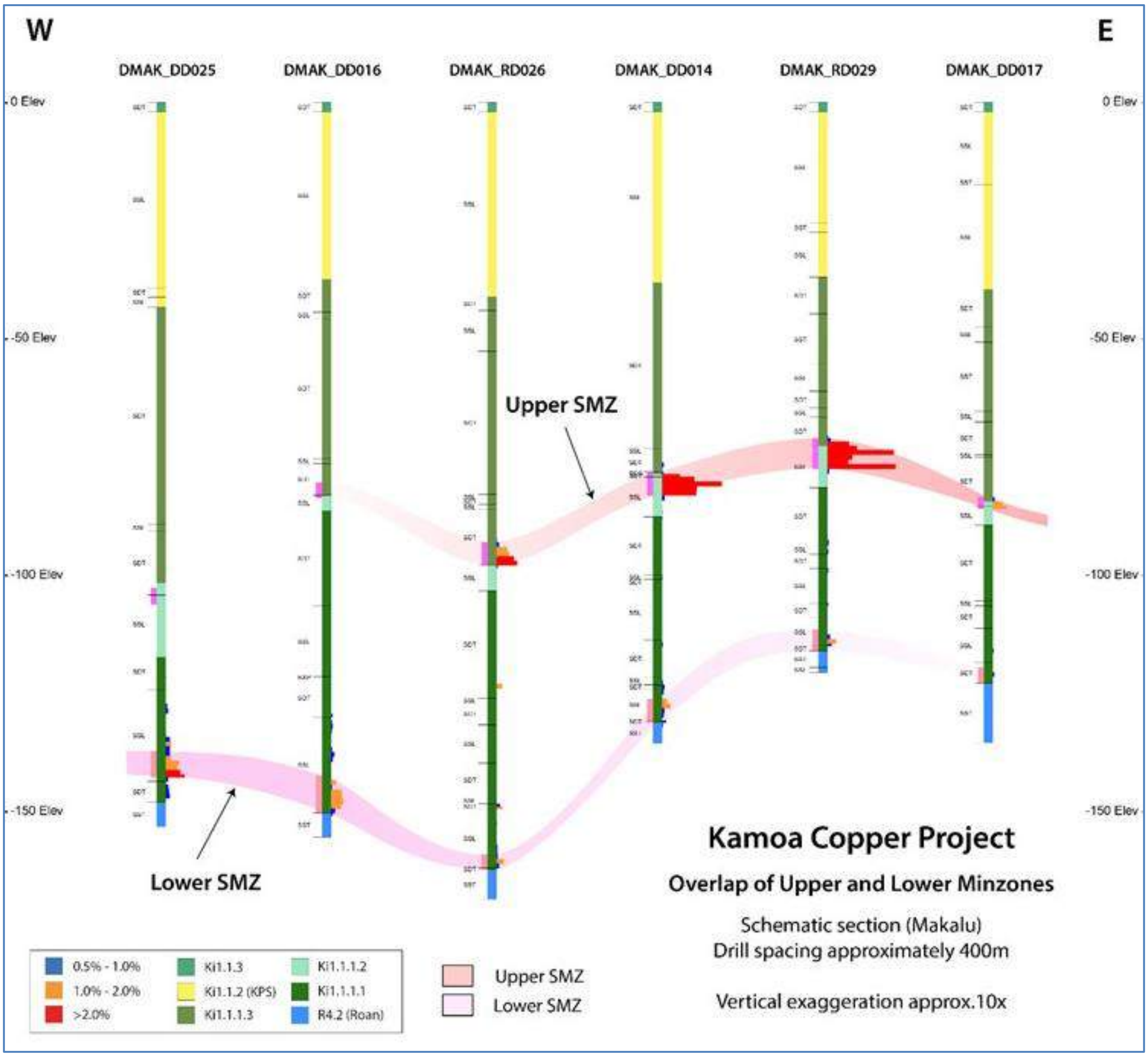


Figure provided by Ivanhoe, 2017. Section line labelled Makalu on Figure 14.1.

14.2.2 Kakula

At Kakula, the highest copper grades are located just above the Roan contact in the basal siltstone. Grades usually drop sharply in the overlying diamictite, and in general, decrease gradually with increasing elevation. For resource estimation, the mineralised zone was defined using an approximate 1% TCu cut-off. No minimum thickness criteria were applied during coding of the mineralised zone, but a minimum 3 m vertical thickness was required during reporting or tabulation of the Mineral Resources to reflect the minimum underground mining height.

Collar, survey, assay, stratigraphy and SG data were exported from the acQuire database in csv format, and then imported into Datamine, where the csv files were combined into a de-surveyed drillhole file. The drillhole file was exported to Excel, and the SMZ selections in each hole were manually selected. The assay file, with the selected SMZs was then imported into Datamine where it was re-combined with the collar and survey files. The model extends from the top of the second siltstone (SSL20) to 5 m below the Roan contact.

14.3 Domaining

14.3.1 Kamoa

Estimation domains at Kamoa were developed by combining the geological and mineralisation models using the stratigraphic and SMZ coding to create domains that honours both the vertical and lateral controls on mineralisation. Eleven domains are modelled (Table 14.1 and Figure 14.4). Contacts between domains are treated as hard contacts for resource estimation purposes.

Table 14.1 Domains Used for Grade Estimation with SMZ Domains Highlighted

Domain	Description
100	Ki1.1.2 (KPS) outside of any portions occurring within the Upper SMZ.
110	Portion of the Upper SMZ occurring within the KPS.
200	Where the Upper SMZ occurs in the Ki1.1.1, this is the zone between the top contact of the Upper SMZ and the base of the KPS (effectively represents unmineralised Ki1.1.1.3).
210	Portion of the Ki1.1.1.2c outside of the Upper SMZ.
220	Portion of the Ki1.1.1.2b outside of the Upper SMZ (very limited in its extent, in close proximity to growth fault/s at Kansoko Sud).
230	Portion of the Ki1.1.1.2a outside of the Upper SMZ (very limited in its extent, in close proximity to growth fault/s at Kansoko Sud).
300	The Upper SMZ where it occurs in the Ki1.1.1.
310	The second Upper SMZ to model the upper portion of the bimodal grade distribution in the Kansoko Sud area.
400	The zone between the bottom contact of the Upper SMZ and the top of the Lower SMZ or top of the Roan (effectively represents unmineralised Ki1.1.1.1).
500	The Lower SMZ.
600	A consistent 5 m thick portion of the uppermost part of the Roan (R4.2).

Figure 14.4 Schematic Illustrating the Vertical Position of the Estimation Domains (Localised Domain 220 and Domain 230 Excluded)

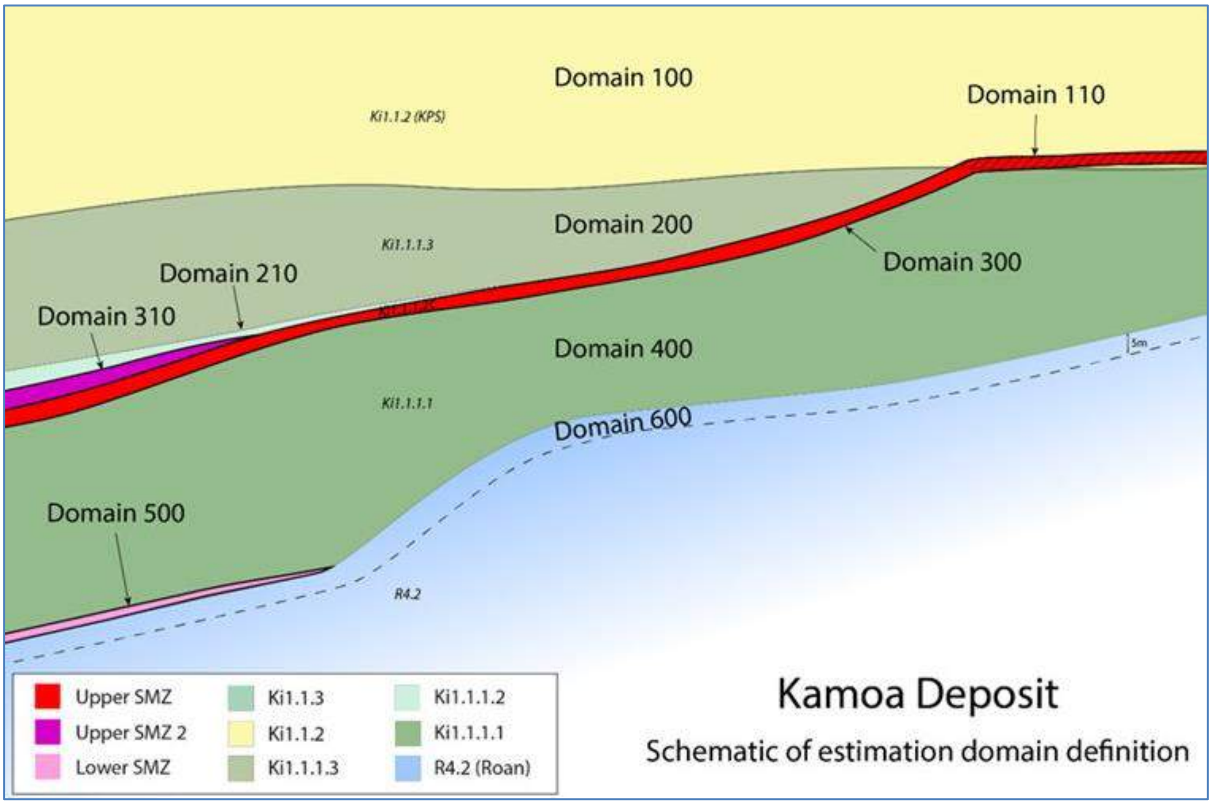


Figure provided by Ivanhoe, 2017.

14.3.2 Kakula

The individual lithological units were combined with the mineralised zone to form six domains used for resource estimation (Table 14.2 and Figure 14.5). The drillholes were first composited to 1 m intervals, and then assigned a domain code using the domain codes in assigned to the block model.

In addition to the six domains, three lateral sub-domains were established to adjust the anisotropy of the search ellipse used for resource estimation to follow the trends of the mineralisation more precisely. Search ranges were elongated along the 115° azimuth in the south-east portions of Kakula, and along a 105° azimuth in the western portion of Kakula and eastern portion of Kakula West. In Kakula West, the mineralisation trends vary locally, and the orientation of the search ranges were adjusted locally using dynamic anisotropy.

Table 14.2 Kakula Domains Used for Grade Estimation with SMZ Domains Highlighted

Domain	Description
440	Second modelled siltstone up from the Roan contact
460	Unmineralised clast-poor diamictite; can pinch out where the full diamictite interval is mineralised
480	Mineralised portion of the diamictite above the basal siltstone. Domain is a combination of the lithology volume and mineralised zone volume
500	Basal siltstone; the primary host to mineralisation
520	Clast-rich, oxidised diamictite. Thin, but persistently developed.
600	A consistent 5 m thick portion of the uppermost part of the Roan (R4.2).

Figure 14.5 Kakula: Vertical Domain Definition

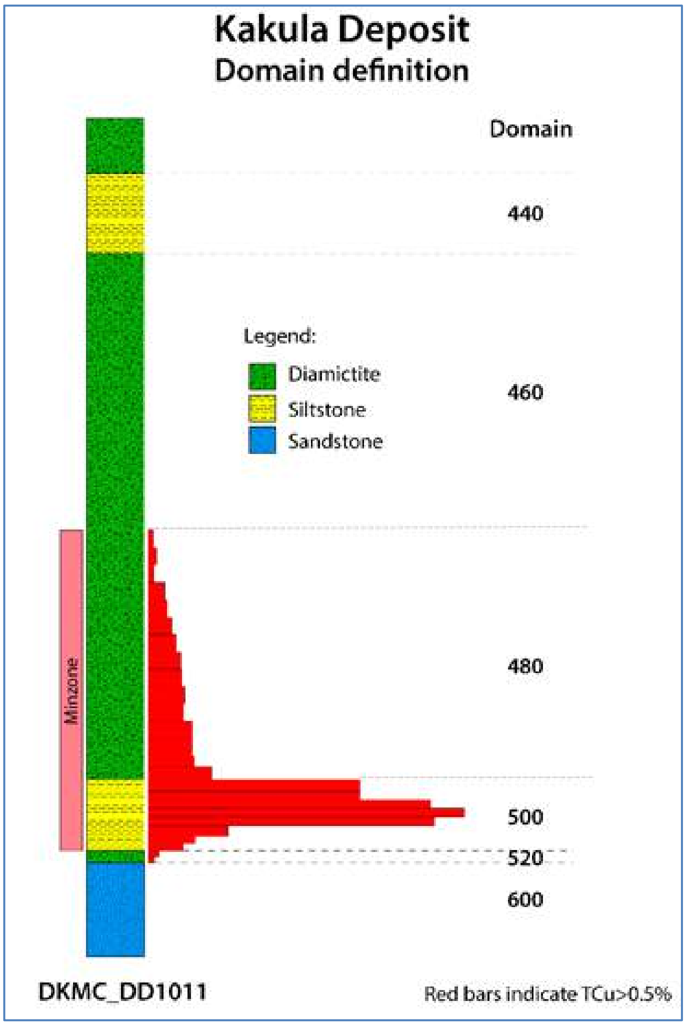


Figure provided by Ivanhoe, 2019.

14.4 Top Capping

14.4.1 Kamoa

Samples are collected at a nominal 1 m sample length, with shorter intervals used to honour geological contacts. Outside of mineralised zones within the KPS, a 3 m composite sample has been submitted for analysis, primarily to determine the sulphur content within the KPS. The drillhole samples were first combined into 1 m composites, honouring domain contacts, and then capped.

Determination of the capping value was based on the following factors:

- Analysis of the histogram and log probability plots for each variable to determine where the grade distribution began to break apart.
- The coefficient of variation (CV), a measure of the standard deviation of the distribution divided by the mean, was assessed. CVs in excess of 1.3 were considered to represent strongly-skewed distributions, or distributions with significant outliers that would require capping.
- Spatial analysis of higher-grades to determine if these grades were supported by other nearby high-grade samples both vertically and laterally.
- A lower capping threshold (as a proportion of the distribution) was applied to domains with limited data.
- Spatial review of the highest TCu grades showed the highest grades are typically clustered and show good connectivity between drillholes. As a result, they were not capped, or had a light capping applied.

Although the TCu CV for most mineralised domains is <1.0 (considered to be low), isolated high-grade samples may have too much local influence during estimation, and were capped. Top capping values were applied per domain prior to estimation. Table 14.3 shows the capping threshold, and the impact of top capping. Top capping was not applied in Domain 310, as the domain is very constrained in its lateral extent, and high-grades are supported by neighbouring drillholes.

Table 14.3 Kamoa: Impact of Top Capping Per Domain on 1 m Composite Samples

Domain	Number of Samples	Capping Grade TCu (%)	Samples Capped	No Capping		With Capping	
				Mean (%)	CV	Mean (%)	CV
100	18,284	4.0%	6	0.05	3.97	0.05	3.60
110	441	10.0%	4	2.71	0.76	2.67	0.70
200	7,366	2.6%	7	0.14	1.85	0.14	1.77
210	1,356	2.5%	2	0.15	1.86	0.14	1.55
300	4,280	18.0%	6	2.56	0.91	2.56	0.90
400	8,969	2.0%	11	0.35	0.86	0.35	0.85
500	291	3.0%	3	0.93	0.78	0.91	0.72

14.4.2 Kakula

At Kakula, top capping was evaluated using 1 m composites within the mineralised zone to assess if isolated high-grades samples existed, and whether these values should be capped to prevent over-estimation.

Kakula is characterised by its high-grade chalcocite-dominant mineralogy. Visual review of the higher-grade composites clearly showed that the higher-grade material holds together laterally along a 115° trend in the south-east portion of the deposit, along a 105° trend in the central portion of the deposit and along a 065° trend in the western portion of the deposit, and is constrained vertically by the basal siltstone (Figure 14.6). In addition, histograms and log probability plots for the mineralised siltstone (Domain 500) show little breakdown in the grade distribution at higher-grades, and the distribution has a low CV value of approximately 0.75. TCu variograms have a low relative nugget effect (10%) and long ranges (2,000 m or longer) along the 115°, 105° and general 065° trends. Based on the strong support for the continuity of the higher grades, and the modelling constraints used, no top capping was applied to samples used in Domain 500. Top capping applied to the other domains is detailed in Table 14.4.

Figure 14.6 Kakula: Visual Top Capping Analyses with TCu grades >8%, >10%, 12%, and >14%

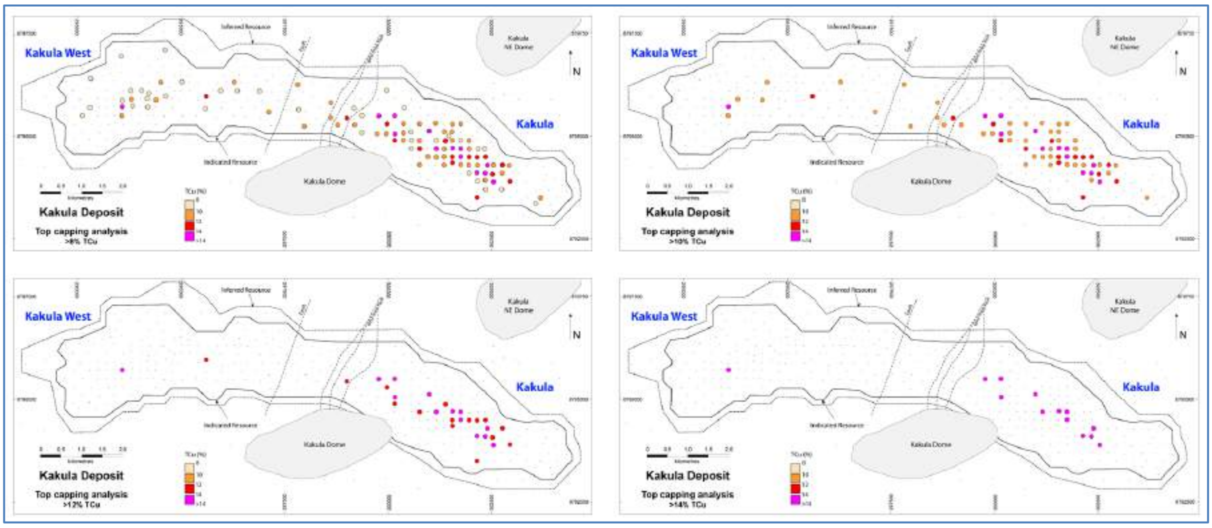


Figure provided by Ivanhoe, 2019.

Table 14.4 Kakula: Impact of Top Capping Per Domain on 1 m Composite Samples

Domain	Number of Samples	Capping Grade TCu (%)	Samples Capped	No Capping		With Capping	
				Mean (%)	CV	Mean (%)	CV
460	6,156	1.5	6	0.11	1.85	0.10	1.71
480	3,783	8.0	2	1.31	0.80	1.31	0.80
520	1,235	3.0	6	0.49	1.10	0.48	0.93
600	1,556	2.0	4	0.09	2.84	0.08	2.44

14.5 Exploratory Data Analysis (EDA)

14.5.1 Kamoa

Composite statistics for 1 m samples for TCu and S are summarised in Table 14.5 and displayed graphically as histograms and log probability plots in Figure 14.7 and Figure 14.8. The distribution of TCu grades within the mineralised zones is positively skewed, but generally well constrained with few outliers. Higher grades are generally clustered, and honour lithological or structural controls.

Table 14.5 Kamoa: 1 m Composite Statistics for Each Domain Option (Uncapped Data)

Variable	Domain	Number of samples	Minimum	Maximum	Mean	Standard Deviation	CV
TCu (%)	100	18,293	0.00	8.42	0.05	0.18	3.86
	110	**441**	**0.03**	**19.40**	**2.71**	**2.05**	**0.76**
	200	7,412	0.00	4.54	0.14	0.26	1.84
	210	1,373	0.00	4.16	0.15	0.23	1.60
	220	397	0.00	1.31	0.12	0.18	1.49
	230	123	0.04	1.66	0.37	0.26	0.72
	300	**4,246**	**0.01**	**22.72**	**2.58**	**2.33**	**0.90**
	310	**238**	**0.00**	**18.15**	**1.76**	**2.47**	**1.40**
	400	9,022	0.00	4.87	0.35	0.30	0.86
	500	**300**	**0.03**	**4.99**	**0.93**	**0.72**	**0.77**
	600	4,072	0.00	1.40	0.04	0.09	1.93
S (%)	100	15,803	0.003	26.09	7.20	5.84	0.81
	110	**433**	**0.003**	**17.52**	**3.89**	**3.04**	**0.78**
	200	6,050	0.003	14.68	0.68	1.00	1.47
	210	1,221	0.003	16.45	1.71	3.09	1.81
	220	396	0.003	7.92	0.56	0.98	1.77
	230	121	0.003	3.72	0.19	0.36	1.86
	300	**3,969**	**0.003**	**22.33**	**1.61**	**1.66**	**1.03**
	310	**237**	**0.010**	**16.61**	**3.28**	**3.72**	**1.14**
	400	8,497	0.003	1.70	0.11	0.12	1.07
	500	**237**	**0.003**	**1.18**	**0.16**	**0.17**	**1.06**
	600	3,457	0.033	3.10	0.05	0.12	2.59

Note: Mineralised domains are in bold.

Figure 14.7 Kamoa: Histograms of 1 m Composites for TCu (%) for All Mineralised Domains

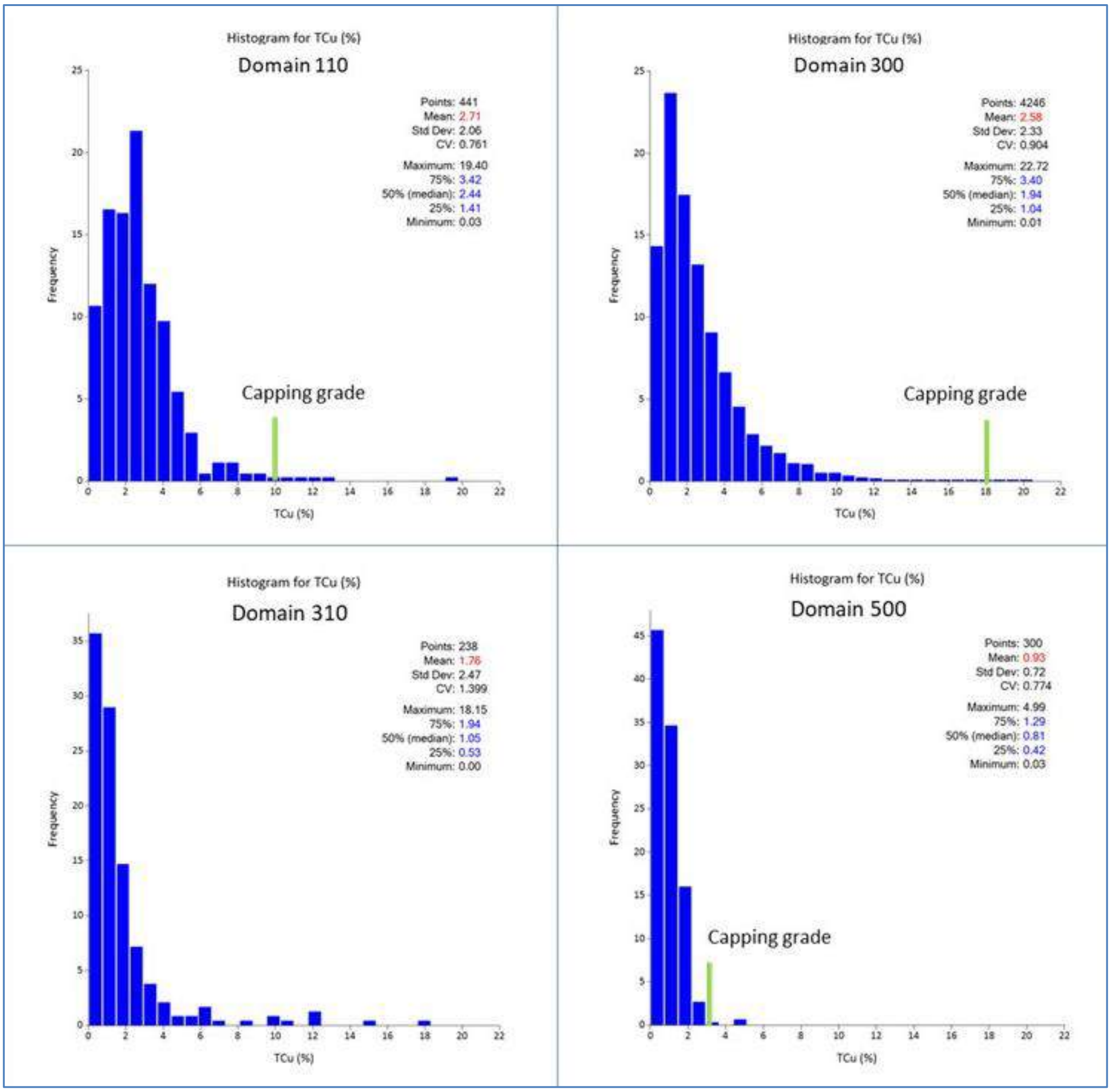


Figure provided by Ivanhoe, 2017. Green lines represent the top capping applied per domain, no capping was applied to Domain 310.

Figure 14.8 Kamoa: Log Probability Plots of 1 m Composites for TCu (%) for All Mineralised Domains

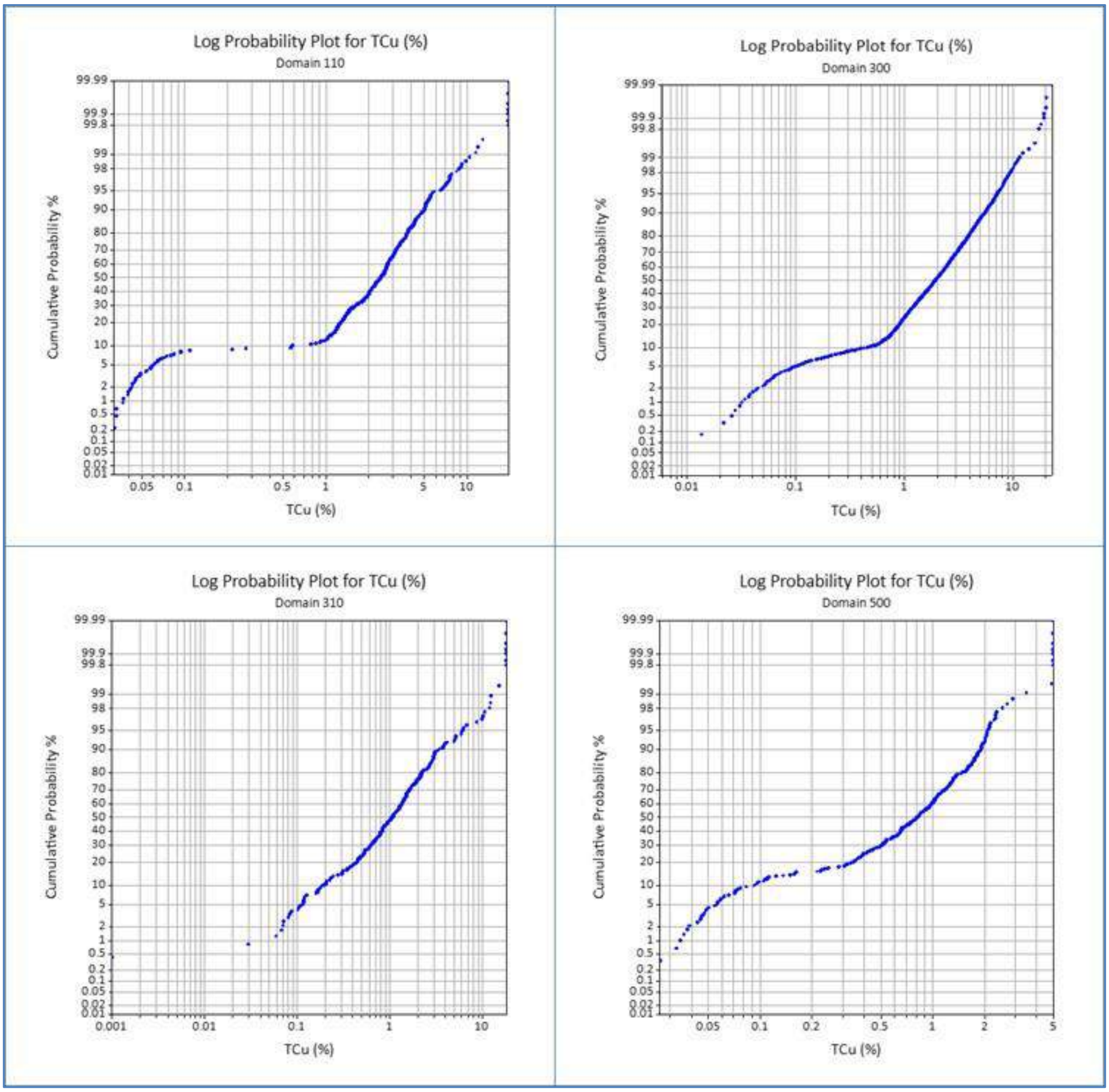


Figure provided by Ivanhoe, 2017.

SG values show very little variability, with distributions approximating normal distributions. Distributions per domain are slightly offset relative to one another depending on the dominant lithology of the domain. The KPS (Domain 100) is primarily shale, with an average SG of 2.79. Domains 200 to 500 are hosted within diamictite or intercalated siltstones, with average SG values of 2.60 to 2.67. The Upper SMZ (Domain 300) has a slightly higher SG of 2.73, likely due to the denser sulphide mineralisation. The porous R4.2 sandstone (Domain 600) has the lowest average SG of 2.51.

Sulphur grades are elevated in the Ki1.1.2 due to high concentrations of pyrite within the siltstone (refer to Table 14.5). Sulphur grades are also elevated in the mineralised domains, most notably domains 110 and 310 where chalcopyrite dominates. A variety of sulphide species occur within Domain 300 with bornite and chalcocite lowering the overall sulphur grade. Domain 500 is chalcocite-dominant; hence the lower sulphur grades. Overall, sulphur values are positively skewed (Figure 14.9). Arsenic values at Kamoa are very low, with approximately 65% of samples <0.001% As (Figure 14.9). In general, most samples have a ratio of ASCu/TCu of 10% or less (representative of sulphides where a small amount will dissolve in sulphuric acid), and very few have a ratio of over 30%, which would normally trigger selection of reagents that would coat the copper oxide minerals to make them float.

Figure 14.9 Kamoa: Histograms and Log Probability Plots for Sulphur and Arsenic Values for Mineralised Domains (Domains 110, 300, 310 and 500)

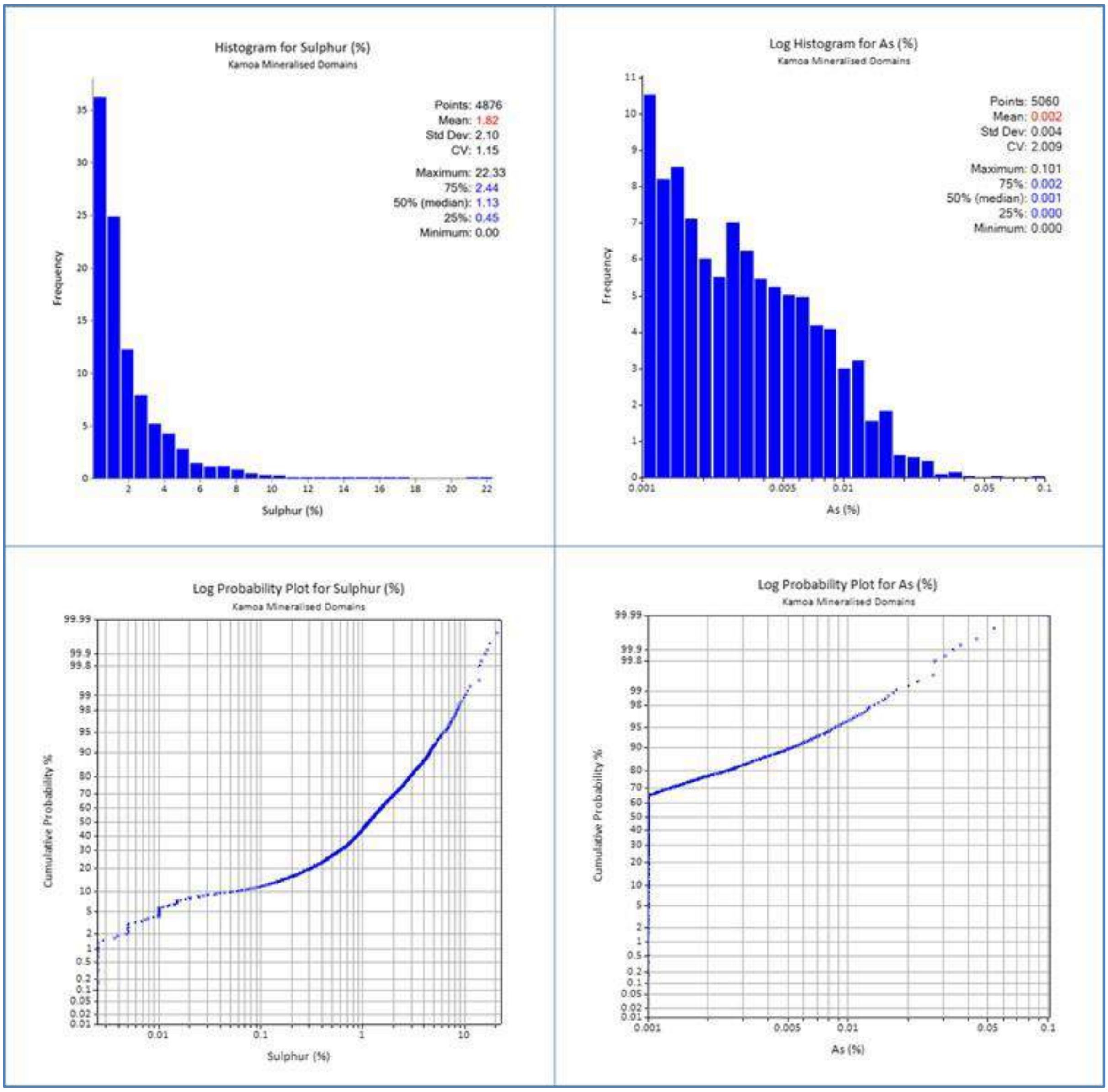


Figure prepared by Ivanhoe, 2017.

14.5.2 Kakula

Exploratory data analysis was undertaken on 1 m composite samples per domain. A total of 155 drillholes were used in the Kakula modelling, and 168 drillholes were used in the Kakula West modelling.

Domain composite summary statistics for TCu and S (%) for each domain are presented in Table 14.6. Histograms and log probability plots for the 1 m composites for the two mineralizedmineralised domains (Domain 480 and Domain 500) are shown in Figure 14.10 and Figure 14.11.

Table 14.6 1 m Composite Statistics for each Domain at Kakula (uncapped data)

Variable	Domain	Number of samples	Minimum	Maximum	Mean	Standard Deviation	CV
TCu (%)	440	3,742	0.00	2.51	0.07	0.25	3.40
	460	6,172	0.00	5.25	0.11	0.20	1.85
	480	**3,789**	**0.00**	**10.53**	**1.31**	**1.05**	**0.80**
	500	**1,215**	**0.02**	**17.74**	**4.78**	**3.57**	**0.75**
	520	1,237	0.00	8.15	0.49	0.54	1.10
	600	1,561	0.00	4.96	0.09	0.25	2.84
S (%)	440	789	0.00	9.19	0.35	0.50	1.43
	460	3,176	0.00	3.77	0.41	0.41	1.01
	480	**3,460**	**0.00**	**4.64**	**0.61**	**0.61**	**1.01**
	500	**1,212**	**0.01**	**7.33**	**1.41**	**1.02**	**0.72**
	520	1,192	0.00	2.95	0.17	0.23	1.41
	600	767	0.00	1.30	0.04	0.09	2.09

Note: Mineralised domains are in bold.

Figure 14.10 Kakula: 1 m Composite TCu (%) for the mineralised basal siltstone (Domain 500). Histogram and Probability Plot

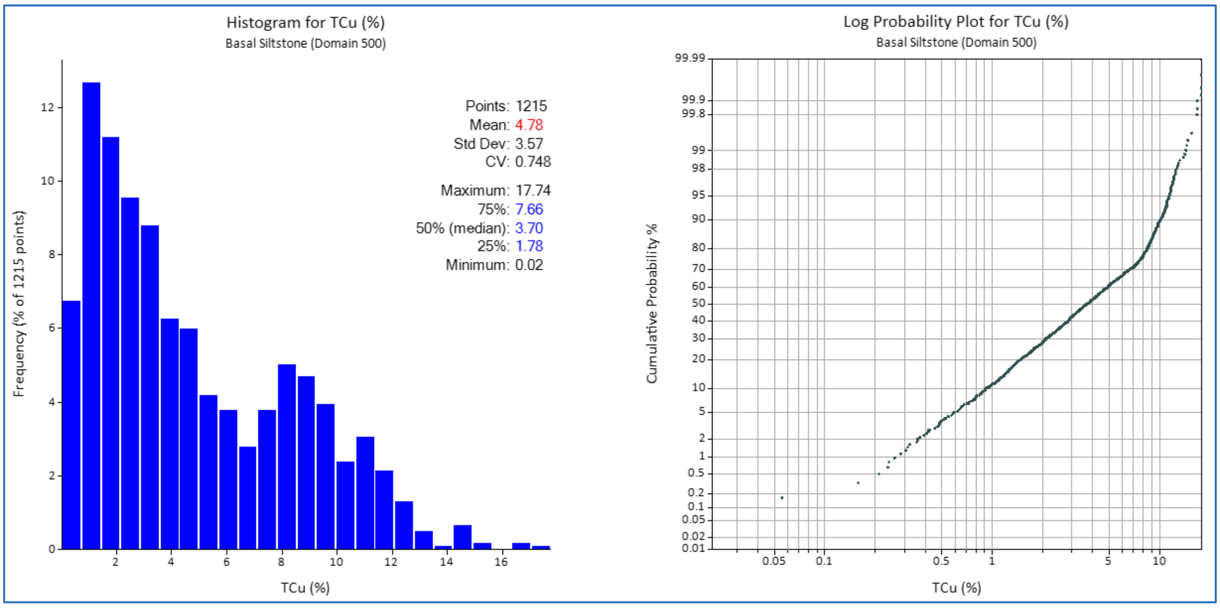


Figure provided by Ivanhoe, 2019.

Figure 14.11 Kakula: 1 m Composite TCu (%) for the mineralised diamictite (Domain 480). Histogram and Probability Plot

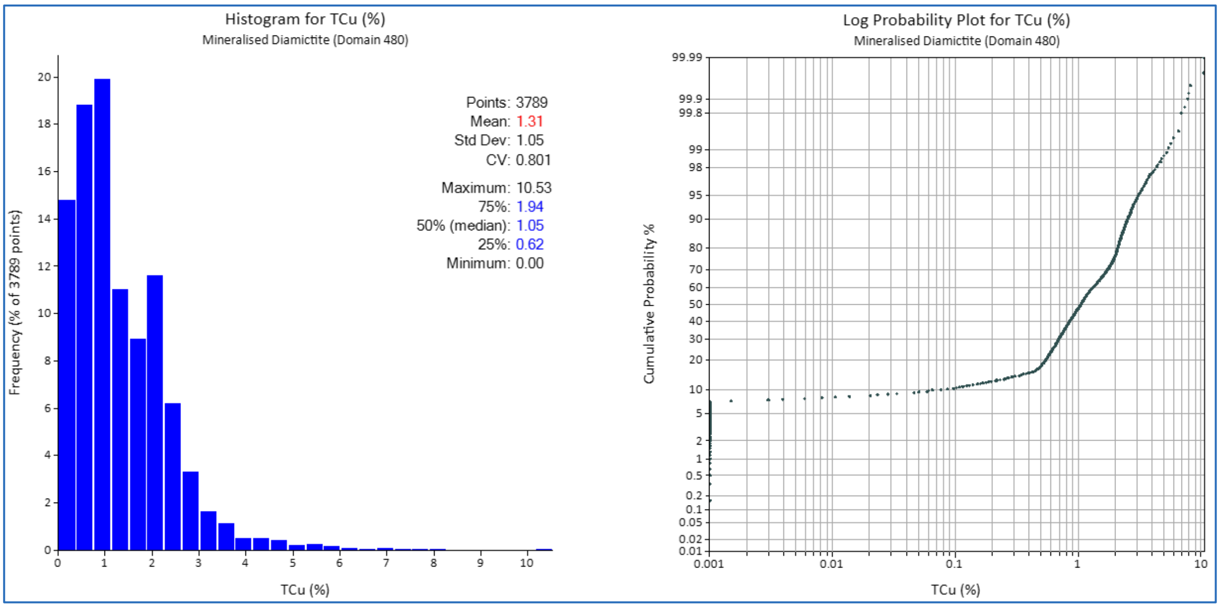


Figure provided by Ivanhoe, 2019.

Higher SG values in the higher-grade zones were recognised early on in the Kakula exploration programme, and SG measurements were collected on whole core for each sample interval that was assayed. Initial holes (prior to DKMC_DD1002) lack a full set of SG data. Since there is a strong relationship between TCu grade (%) and SG, a regression was performed (Figure 14.12) and used to assign an SG value to those samples with missing SG values.

Figure 14.12 Kakula Scatter Plot of Total Copper (%) and SG Values for the 3% TCu Grade Shell (SMZ30), with Outliers Removed

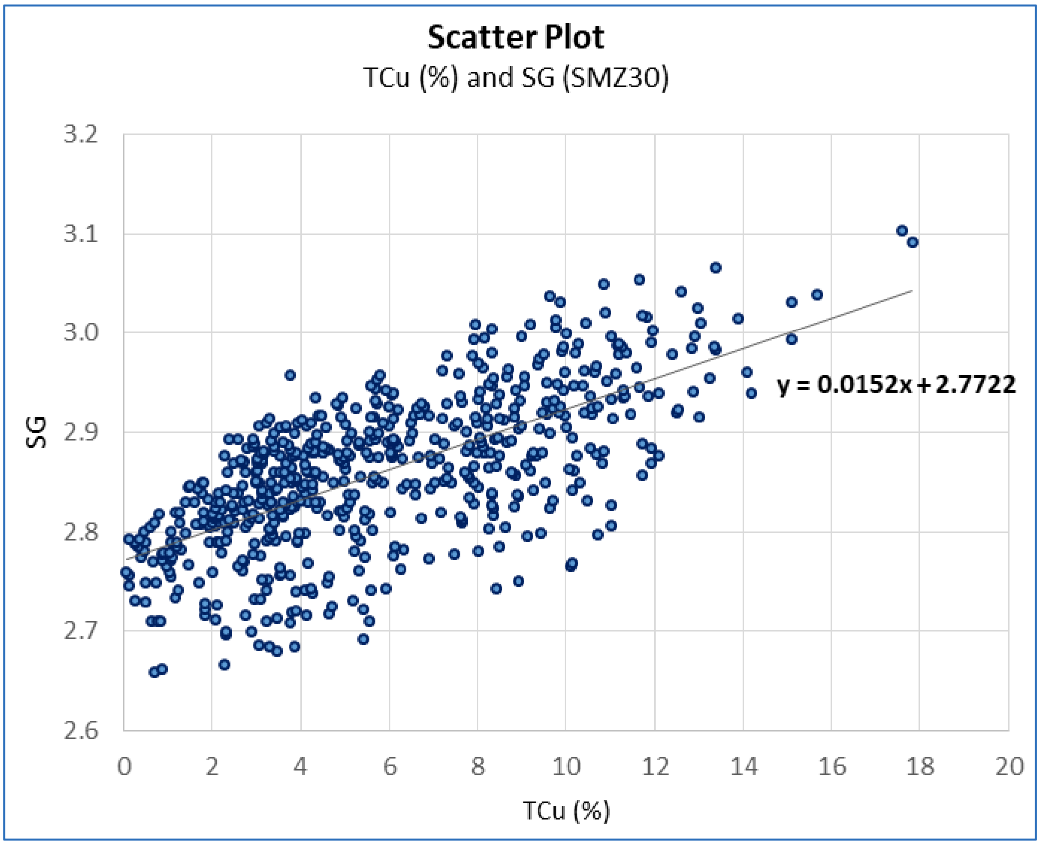


Figure provided by Ivanhoe, 2017.

14.6 Statistics Observations

14.6.1 Kamoa

Within the mineralised domains, TCu grades (refer to Figure 14.7) are positively skewed, but well constrained.

No clear relationship is evident between TCu and ASCu. Higher ASCu grades are usually highly localised and concentrated in only one or two drillholes, indicating an inability to distinguish sulphide and oxide mineralisation into separate domains. Geological and metallurgical studies of the sulphide species indicate that the bulk of the mineralisation at Kamoa is sulphide, with localised oxide mineralisation closer to surface and along dome edges.

SG values for individual domains approximate a normal distribution, or are very weakly negatively skewed, with the mean SG value per domain strongly influenced by the dominant host lithology. The CV for individual domains is low, typically 0.1 or lower.

14.6.2 Kakula

By nature of the domain definition and construction, TCu grades are well constrained vertically. The weak bimodality of the 1 m composite samples within Domain 500 is a result of very high-grade central portion of the deposit being surrounded by lower-grade material. The bimodality is accounted for in the resource estimation by alligning the anisotropy of the search ranges and variography with the trends of the high-grade material.

14.7 Structural Model

Eight structures were defined at Kamoa using geophysical data, and lithological discontinuities interpreted from the drillhole data (refer to Section 7.3.4. These structures were then used to divide the model into nine structural zones with internally similar strikes and dips. For grade estimations, the blocks and drillholes were transformed to two dimensions representing each structural zone, with the SMZs allowed to be included across the structural domain boundaries in the estimation. The same approach was adopted at Kakula, with five structural blocks defined.

Amec Foster Wheeler concurs that this approach is reasonable, as the faulting in most of the Project area appears to have occurred after the deposition of the mineralisation.

Currently, it is difficult to establish the dips of the interpreted faults, and/or to determine if they are a single fault plane or represent a fault zone. For the Kamoa resource model, the simplest interpretation of the faults was used, which assumed that the faults are single vertical planes. Fault intervals identified in drill core at Kakula have allowed a steep dip (approximately 75°) to be modelled for these faults. Other faults and/or fractured zones have been mapped, based on geophysics and observed broken core; however, the available data are too wide-spaced to establish the dip and extent of these faults. Structural information from the initial mine development drives at Kakula should be evaluated and included in future resource estimates as it becomes available. This will be a key piece of information in understanding the geometry of the mineralizationmineralisation and its implication on the efficacy of the proposed grade control and mining methods.

14.8 Surface and Block Modelling

14.8.1 Kamoa

Surface modelling and block model estimation were limited within perimeters defining the mineralised portions and permit boundaries of the Project. Two prominent domes, the Kamoa dome to the north and the Makalu dome to the south, were excluded from the modelling as they represent leached areas, or barren areas where the Roan sandstone (R4.2) crops out at surface.

The Mineral Resource area was subdivided into nine structural domains based on the structural model and coded with grade domains using wireframes that define the stratigraphic units and mineralised zones. A 3D block model was established using the parameters provided in Table 14.7.

Table 14.7 Kamoa: Block Model Parameters

Axis	Origin	Maximum	Block Size (m)	No. of Blocks
Easting (X)	300,000	315,000	50	300
Northing (Y)	8,797,000	8,821,000	50	480
Elevation (Z)	-300	1,600	1	1,900

To improve stationarity for grade estimation, both the drillholes and the block model were transformed to ensure that the vertical TCu grade profiles matched between drillholes. Typically, these profiles are bottom-loaded, with the higher-grades occurring at the bottom of the profile and grading upwards to lower and lower-grades towards the top of the profile. The transformation was performed by adjusting the Z-coordinate of the data to 'dilate' drillhole composites and blocks to the maximum vertical thickness of the SMZ for each domain. This ensures that the lower, middle and upper portions of the grade profile correctly align between drillholes (Figure 14.13).

Hard boundaries were used for individual stratigraphic and mineralisation domains (whereby only data within the domain are used), and soft boundaries were used for structural domains. Variography and estimation were completed in transformed space. The block models were transformed back to their original vertical location by setting the centroid of each block back to its original Z-coordinate.

Figure 14.13 Kamoa: Vertical Section Showing Untransformed Composites and Blocks (Top) and Transformed Composites and Blocks (Lower) for Domain 300, 3x Vertical Exaggeration

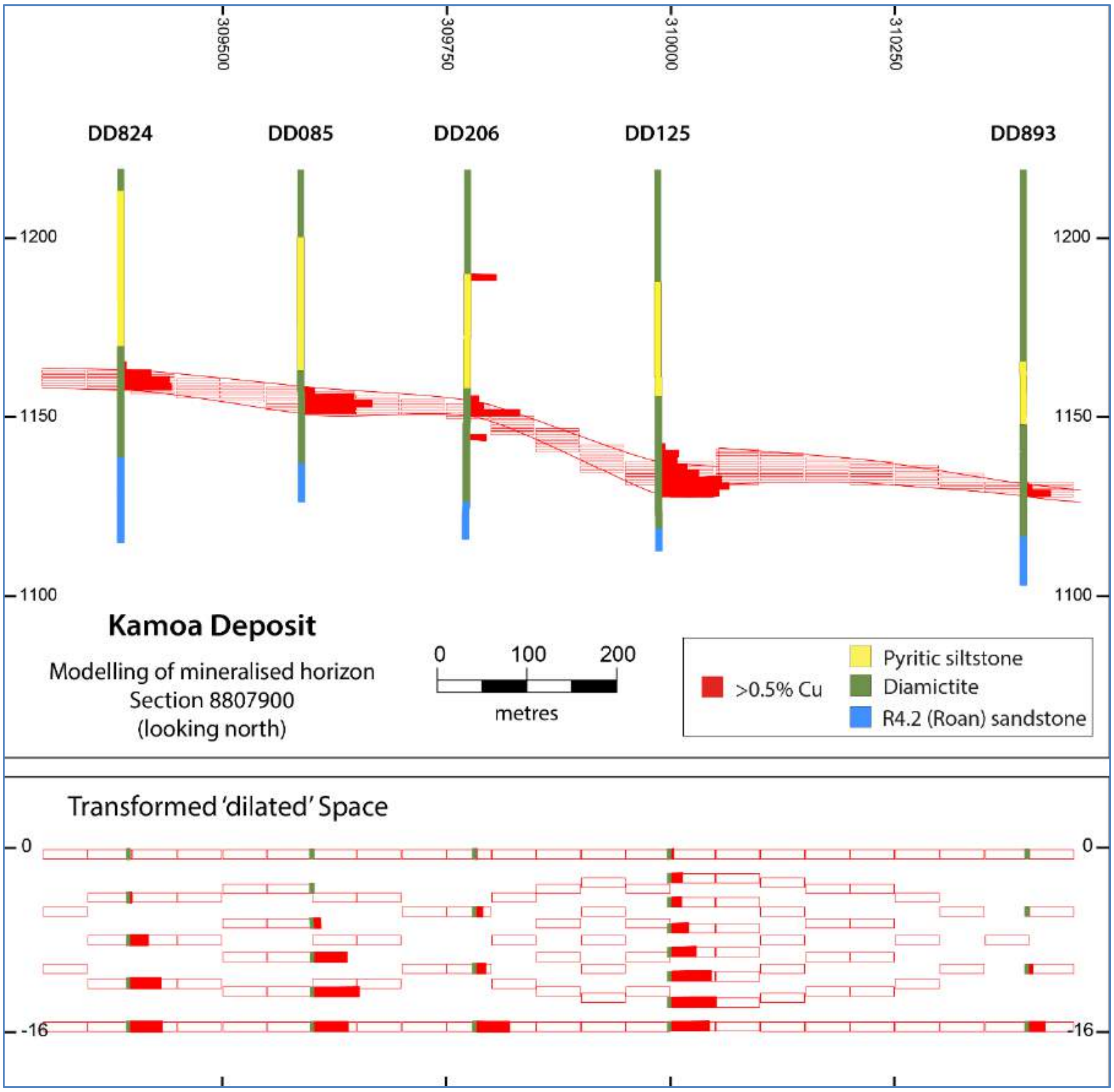


Figure prepared by Ivanhoe, 2019; Copper grade intensity shown by bars on right side of hole.

Transformed 1 m composites were used for variography. The variogram parameters were first optimised by performing sensitivity studies on the lag, angular tolerance, bandwidth and transform applied (normal-score transform) prior to modelling of the variogram. The vertical bandwidth was a key element given the vertical TCu trends evident in drillholes, and was typically set to a narrow interval. Downhole variograms of the transformed 1 m composites were used to determine the nugget effect (C0). The transformation dilates downhole samples, moving them further away from each other and potentially overstating continuity at short ranges. As a validation, downhole variograms of untransformed 1 m composite samples were also investigated and were found to be comparable. Variogram parameters for TCu for mineralised zones are summarised in Table 14.8, and variogram parameters for S for mineralised zones are summarised in Table 14.9.

Weak anisotropy at 115° is evident in Domain 110 (the mineralised portion of the KPS). A more robust, and stronger developed anisotropy at 145° is evident in the Upper SMZ (Domain 300) for both TCu (Figure 14.14) and S. This orientation matches that of the thickness changes evident in the stratigraphic units, and parallels the interpreted orientation of growth faults at Kamoa.

Table 14.8 Kamoa: TCu Variogram Parameters Categorised by Mineral Domain

Domain	Major Direction	Nugget C_0	C_1	Range (m)	C_2	Range (m)	C_3	Range (m)
110	115°	0.09	0.51	545	0.4	2025		
				400		1000		
				3		6		
300	145°	0.05	0.36	300	0.25	530	0.35	2300
				200		300		1100
				13		16		22
310	145°	0.16	0.4	172	0.44	1035		
				100		300		
				3		18		
500	145°	0.11	0.6	600	0.29	1600		
				500		1300		
				69		70		

Note: Domain 500 used variogram parameters from Domain 400.

Table 14.9 Kamoa: S Variogram Parameters Categorised by Mineral Domain

Domain	Major Direction	Nugget C_0	C_1	Range (m)	C_2	Range (m)	C_3	Range (m)
110	120°	0.06	0.49	395	0.46	1900		
				395		1300		
				6		7		
300	145°	0.04	0.47	640	0.49	1600		
				280		1100		
				29		37		
310	135°	0.10	0.66	210	0.24	950		
				200		340		
				10		12		
500	140°	0.11	0.43	320	0.22	1000	0.24	4500
				320		1000		1840
				89		119		143

Note: Domain 500 used variogram parameters from Domain 400.

Figure 14.14 Kamoa: Normal Score Major and Semi-Major Direction Variograms for TCu (Domain 300)

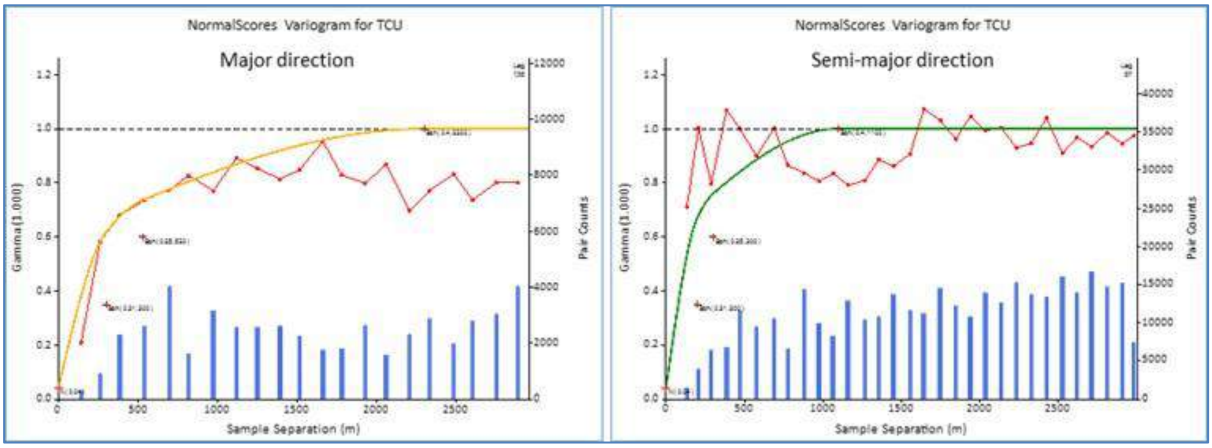


Figure prepared by Ivanhoe, 2017.

All grade variables (TCu, ASCu, As, Fe, and S) were estimated into each block using OK interpolation for reporting. An inverse distance to the second power (ID2) and nearest neighbor (NN) estimate were used for validation purposes. Estimation parameters are summarised in Table 14.10. Search parameters were adjusted for each variable within each domain based on the grade continuity evident from the variography. For all variables, if the block remained unestimated following the first search, the search was doubled in size. If necessary, this was again expanded by a factor of 2.5 for a third search.

Table 14.10 Kamoa: Estimation Parameters for TCu for all Mineralised Domains

Domain	Orientation			Search Range	Number of Samples Search Pass 1		Number of Samples Search Pass 2	
	Axis	Azimuth	Dip		Minimum	Maximum	Minimum	Maximum
110	X	115°	0°	1,000	4	12	4	8
	Y	0°	0°	500	4	12	4	8
	Z	0°	90°	5	4	12	4	8
300	X	145°	0°	1,100	4	12	4	8
	Y	0°	0°	600	4	12	4	8
	Z	0°	90°	5	4	12	4	8
310	X	145°	0°	450	4	12	4	8
	Y	0°	0°	200	4	12	4	8
	Z	0°	90°	5	4	12	4	8
500	X	145°	0°	1,200	4	12	4	8
	Y	0°	0°	1,000	4	12	4	8
	Z	0°	90°	5	4	12	4	8

A limit of a maximum of three samples from a single drillhole was used to ensure that at least two drillholes were used for any estimate, preventing any possible string effect occurring, where weights are preferentially assigned to the outermost samples when all samples used in an estimate are aligned in a row.

ASCu values are not available for every sample that contains a TCu value. This is particularly relevant in the Ki1.1.2, where only 21% of TCu samples have a corresponding ASCu value. Within the Upper SMZ (Domain 300), 94% of TCu samples have a corresponding ASCu value. To overcome this, an OK estimation of TCu and ASCu using the search and variogram parameters for TCu was completed using only samples that contained both a TCu and ASCu value. Using this estimate, the ratio of TCu:ASCu was calculated. The final ASCu grade was then back-calculated from the TCu estimate (using all available TCu samples) and the calculated ratio.

14.8.2 Kakula

Surface elevation modelling and block model creation were limited by perimeters defining the unoxidised mineralised portions of the project. Domes north and south of the deposit were excluded from the resource model as they represent eroded or leached barren areas. The extents of the Kakula models were defined by a rectangle that encloses the existing drillholes.

The Mineral Resource area was subdivided into five structural domains using the Kakula structural model. A wireframe solid was constructed for each stratigraphic unit in Leapfrog Geo software using structural discs representing the local dip and dip direction of the Roan and KPS surfaces created using Datamine software. This was achieved by first constructing an initial digital terrain model (DTM) of the Ki1.1.1–R4.2 contact and the Ki1.1.1–Ki1.1.2 contact in Leapfrog, importing these DTMs into Datamine, and then estimating the dip and dip direction of the DTM surface into structural discs. These structural discs were then imported into Leapfrog where they were used to control the local dip and dip direction of individual stratigraphic units. Control points were also added along the edges of fault blocks and domes in areas of sparse data to ensure relatively consistent offset along the faults. These wireframe solids were then used to code the stratigraphic units into the block model.

A prototype model was established using 50 m x 50 m blocks in easting and northing, with 1 m blocks in elevation using the parameters provided in Table 14.11.

Table 14.11 Kakula Block Model Parameters

Axis	Origin	Maximum	Block Size (m)	# Blocks
Easting (X)	290,000	306,000	50	320
Northing (Y)	8,792,000	8,798,000	50	120
Elevation (Z)	-300	1,500	1	1,800

Estimation occurred in transformed space per domain, using OK. ID3 and NN, estimates were used as validation. Anisotropic searches were aligned at 115° in the south-east portion, 105° in the central portion, and generally in a north-east direction in the western portion (but vary locally) to honour the spatial anisotropy of TCu grades and lithological thicknesses (Table 14.12 and Figure 14.15).

Table 14.12 Kakula: Estimation Parameters Used for the First Search (Domain 500)

Search Domain	Orientation			Search Range	Number of Samples		Estimation Method
	Axis	Azimuth	Dip		Minimum	Maximum	
South-East	X	115°	0	1,000	4	12	OK
	Y	25°	0	400	4	12	OK
	Z	0°	-90	5	4	12	OK
Central	X	105°	0	800	4	12	OK
	Y	15°	0	600	4	12	OK
	Z	0°	-90	5	4	12	OK
Western	X	variable	0	800	4	12	OK
	Y	variable	0	400	4	12	OK
	Z	0°	-90	5	4	12	OK

Figure 14.15 Kakula: Anisotropy Angles Used at Kaula West, Overlain on the Ki1.1.1-R4.2 (Roan) Contact

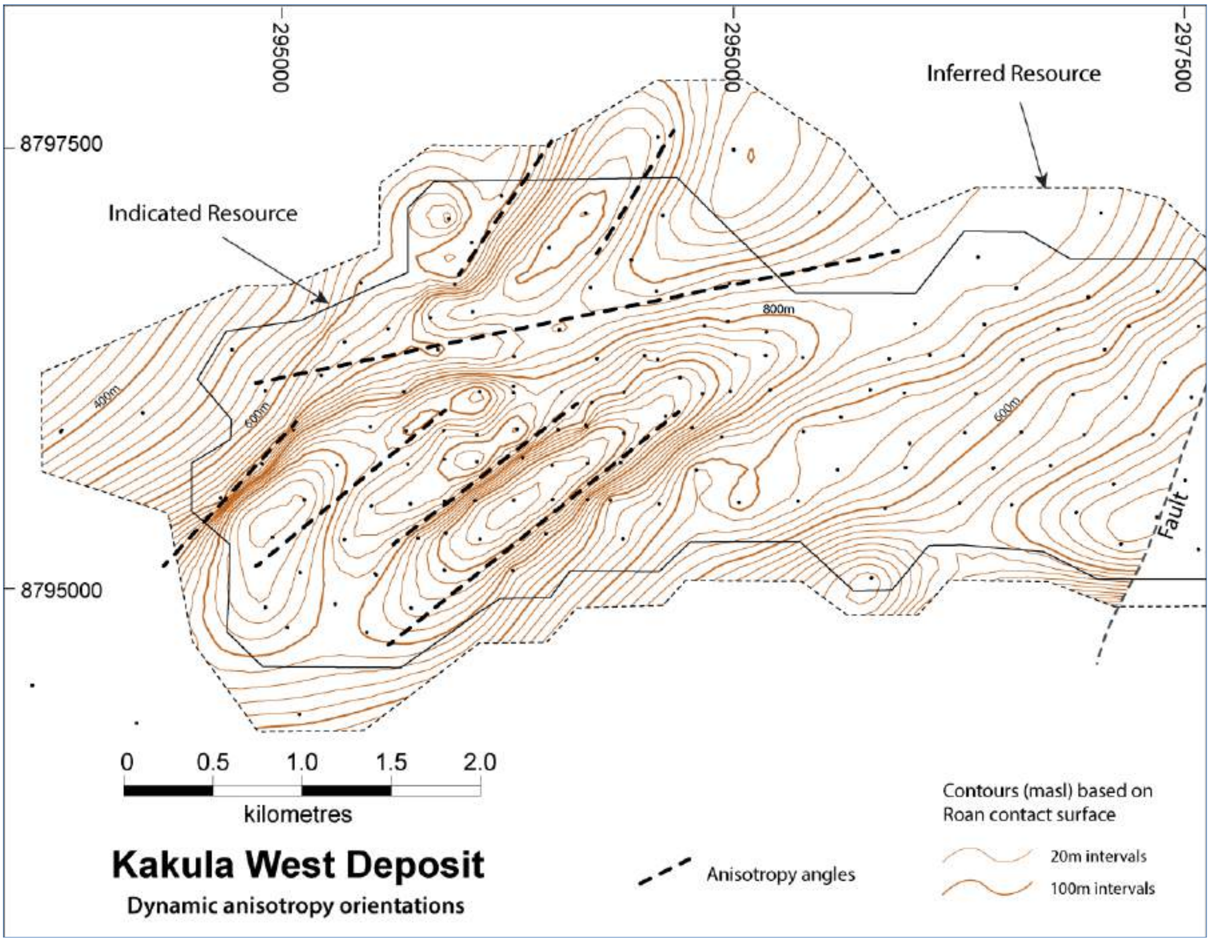


Figure prepared by Ivanhoe 2019.

14.9 Specific Gravity

For Kamoa, SG was estimated in transformed space using ID2, using only those SG samples that occurred within individual domain wireframes (refer to discussions in Section 11.5 on SG determinations). Search parameters were the same as those used for sulphur, with the minimum number of samples and maximum number of samples used in estimation the same as those in Figure 14.10.

For Kakula, SG data were available for the majority of drillhole samples, and regression values were available where sample data were missing (Section 14.5). SG was estimated as a separate variable, using OK with its own search and variogram parameters.

14.10 Mineral Resource Classification

The same drillhole spacing criteria are used at both Kamoa and Kakula to classify Mineral Resources. Areas outlined by core drilling at 800 m spacing with a maximum projection distance of 600 m outward of drill sections, and which show continuity of grade at 1% TCu, geological continuity, and continuity of structure (broad anticline with local discontinuities that are likely faults) were classified as Inferred Mineral Resources over a combined area of 25.3 km^2. Mineral Resources within a combined area of 72.2 km^2 that were drilled on 400 m spacing and which display grade and geological continuity were classified as Indicated Mineral Resources. The total area of the Kamoa-Kakula Project is approximately 410.1 km^2.

The resource classification for Kamoa is shown in Figure 14.16, and for Kakula in Figure 14.17. Although the Kamoa resource model was updated from 2014, all additional drillholes were infill holes within the Indicated outline. Only minor changes to account for adjustments to the shape of the domes have been made to the classification since 2013.

Figure 14.16 Kamoa: Mineral Resource Classification

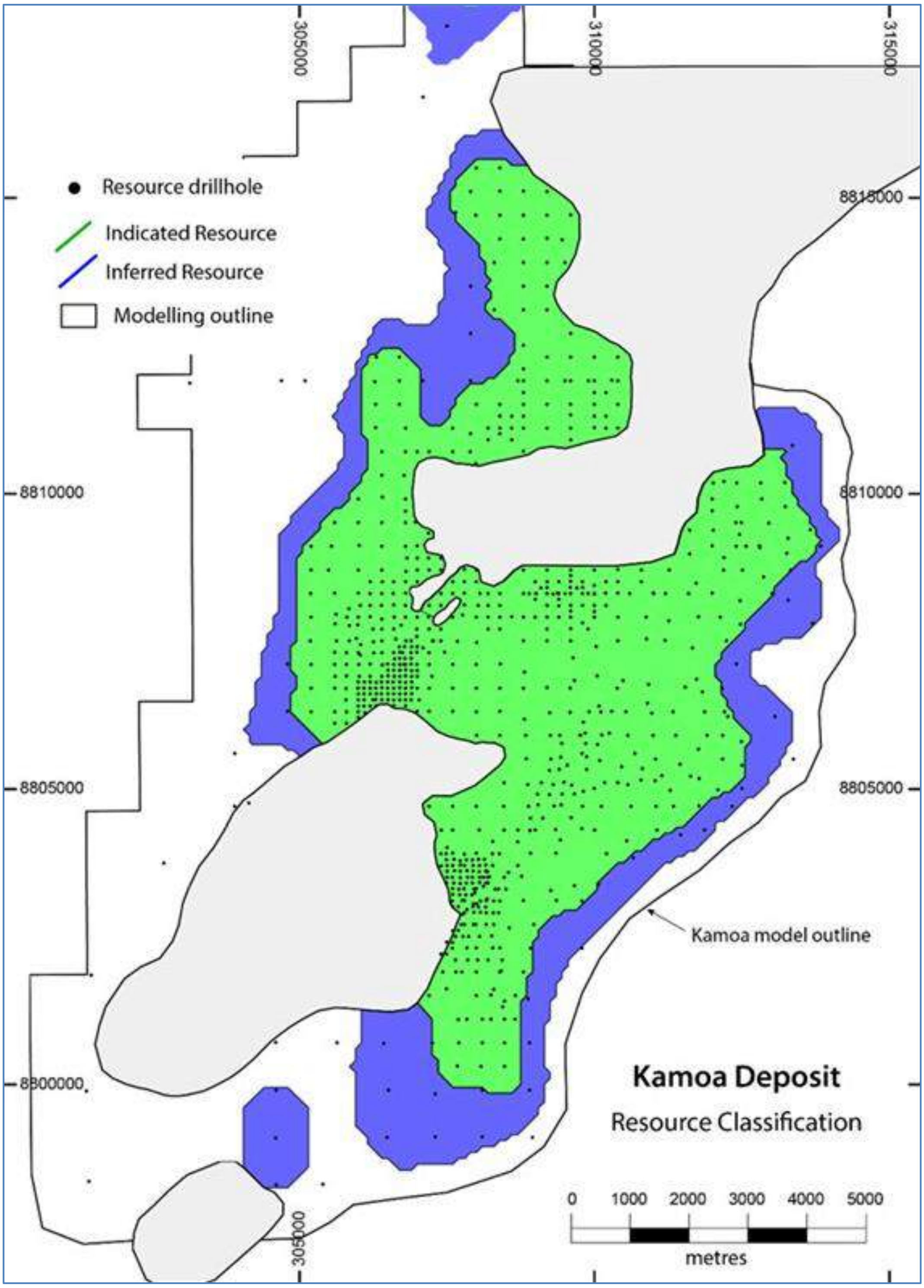


Figure prepared by Ivanhoe, 2017.

Figure 14.17 Kakula: Mineral Resource Classification and Expansion Since 2017

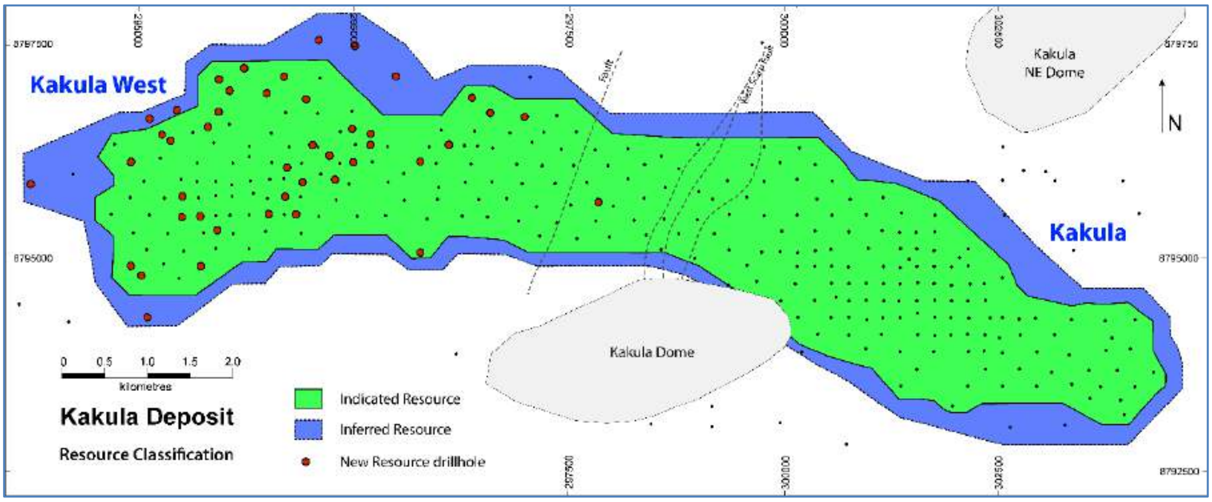


Figure provided by Ivanhoe, 2019.

14.11 Model Validations

14.11.1 Visual Checks

Estimated block grades and composite grades were compared visually in plan view and showed a good agreement, refer to Figure 14.18 for Kamoa, and Figure 14.19 and Figure 14.20 for Kakula.

The previous 2D modelling approach used at Kamoa was updated for the additional drillhole data to allow comparison of the 2D and 3D models. To ensure a fair comparison, the 2D model was trimmed within the Upper SMZ perimeter, and the 3D model blocks were selected within a wireframe defining the upper and lower contacts of the 1.0% 2D model (SMZ10). These blocks were combined to the 2D model block size of 100 m x 100 m (i.e. sets of four 50 m x 50 m blocks were combined together) and a single cell in the vertical representing the vertical thickness.

Figure 14.18 Kamoa: Estimated TCu Grade (%) for the Upper SMZ (Domain 300)

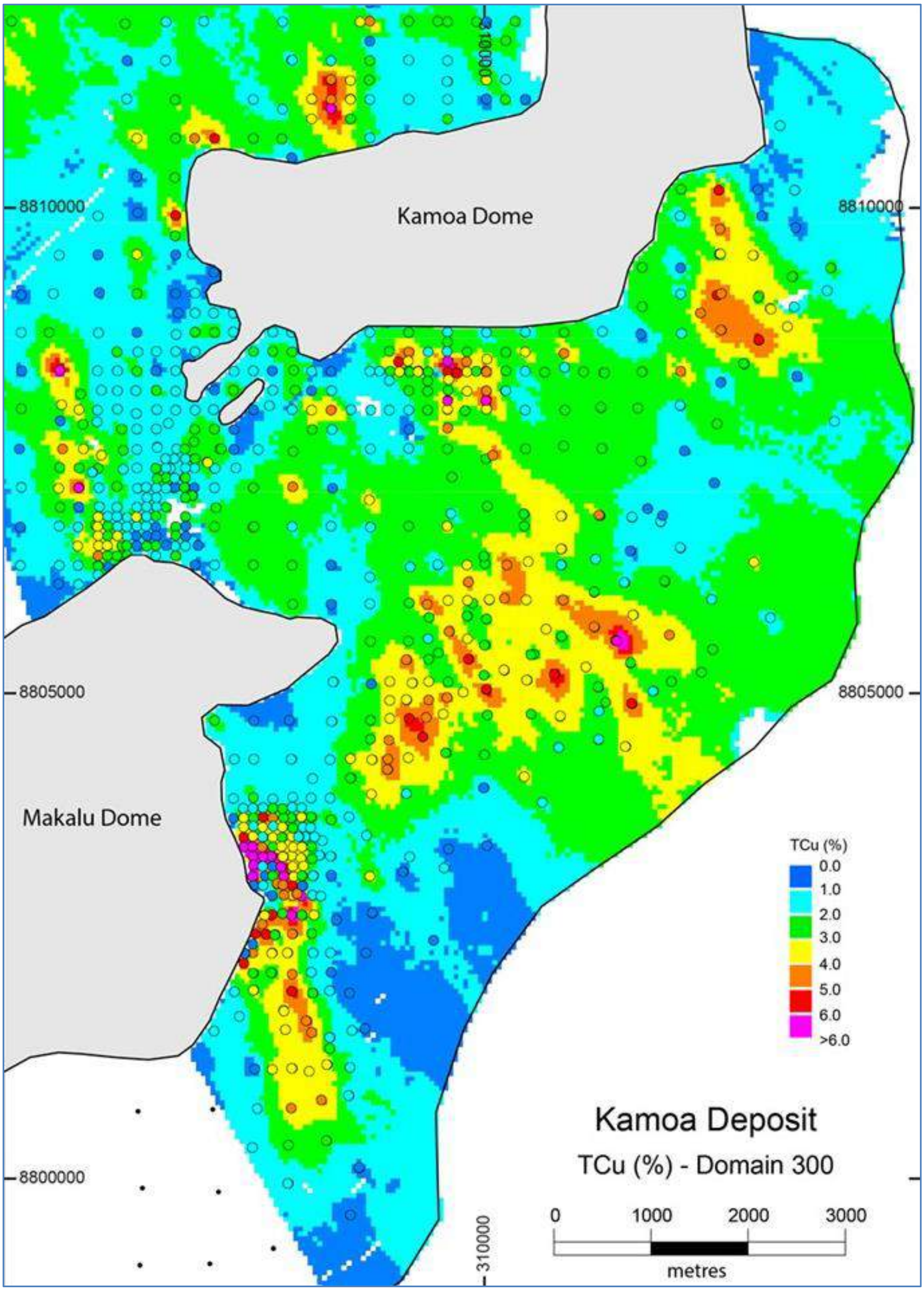


Figure prepared by Ivanhoe, 2017.

Figure 14.19 Kakula: Estimated TCu (%) for the Basal Siltstone (Domain 500)

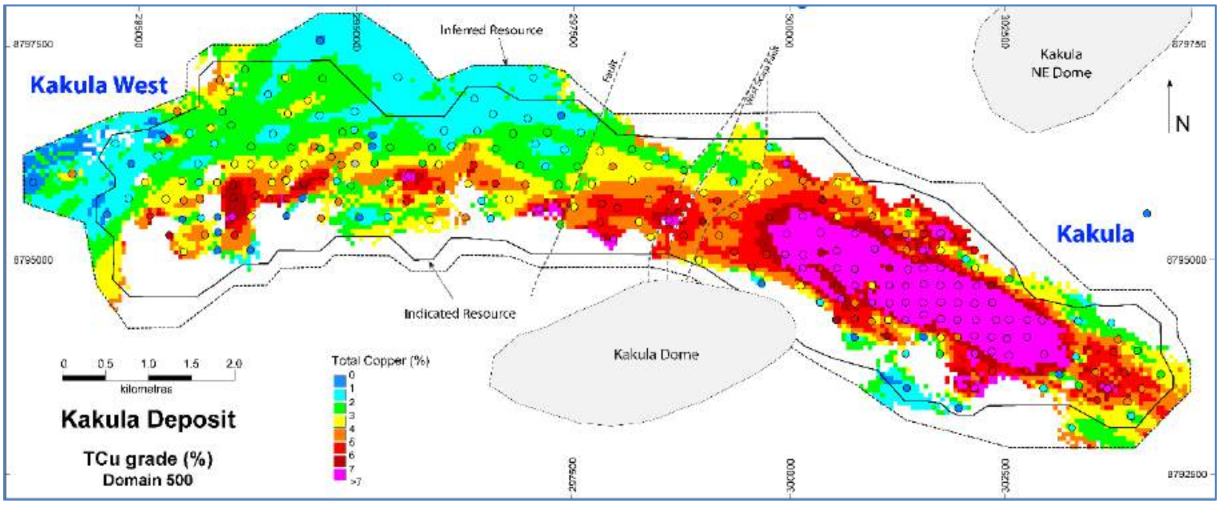


Figure prepared provided by Ivanhoe, 2019. Missing blocks are due to fault offsets.

Figure 14.20 Kakula: Estimated TCu (%) for the Mineralised Diamictite (Domain 480)

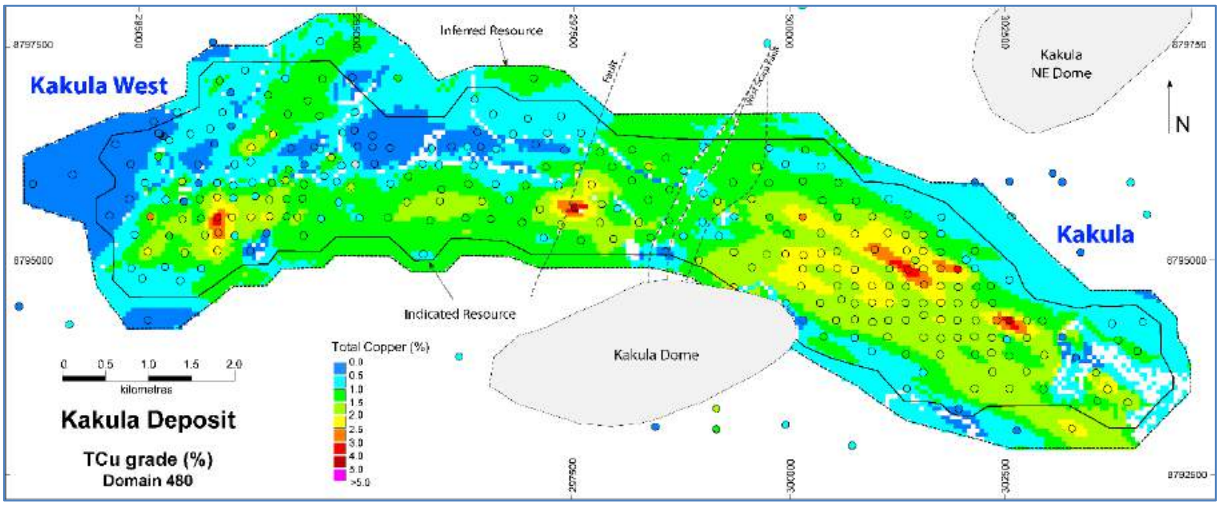


Figure prepared provided by Ivanhoe, 2019. Missing blocks are due to fault offsets.

These models were compared visually and graphically through plotting the grade and tonnage at different cut-offs (Figure 14.21), and by plotting the grade distributions in plan view (Figure 14.22). These comparisons indicate that the SMZ selection and estimation technique have both contributed to a smoothing of the block grades in the 3D model when considered over the full mineralised interval; however, this smoothing is overcome when the vertical definition available in the original 50 m x 50 m x 1 m blocks from the 3D model are used (the stippled lines in Figure 14.21).

Figure 14.21 Comparative Grade-Tonnage Curves for the Kamoa 2D and 3D Models Constrained Within the Upper SMZ (Domains 110, 300, and 310) and Wireframe Defining the 1.0% TCu 2D Model (SMZ10)

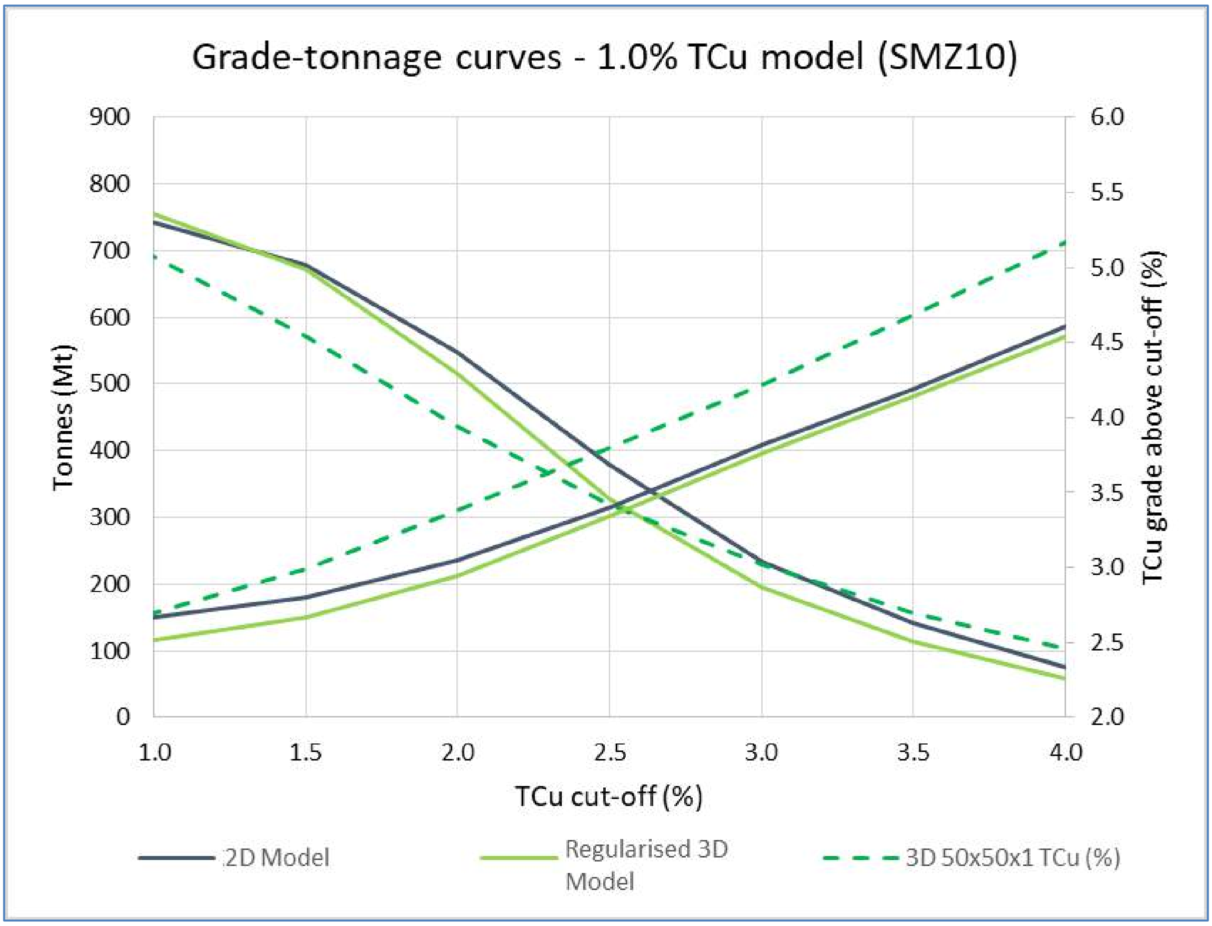


Figure prepared by Ivanhoe, 2018. Solid lines represent two-dimensional model (100 m x 100 m x variable height) and two-dimensional equivalent of the 3D model ('Regularised 3D Model'); stippled line represents reporting from the 3D model (50 m x 50 m x 1 m).

Figure 14.22 Comparison of 3D (left) and 2D (right) Models for Estimated TCu Grade (%) for a 1.0% TCU Modelling Cut-off (SMZ10)

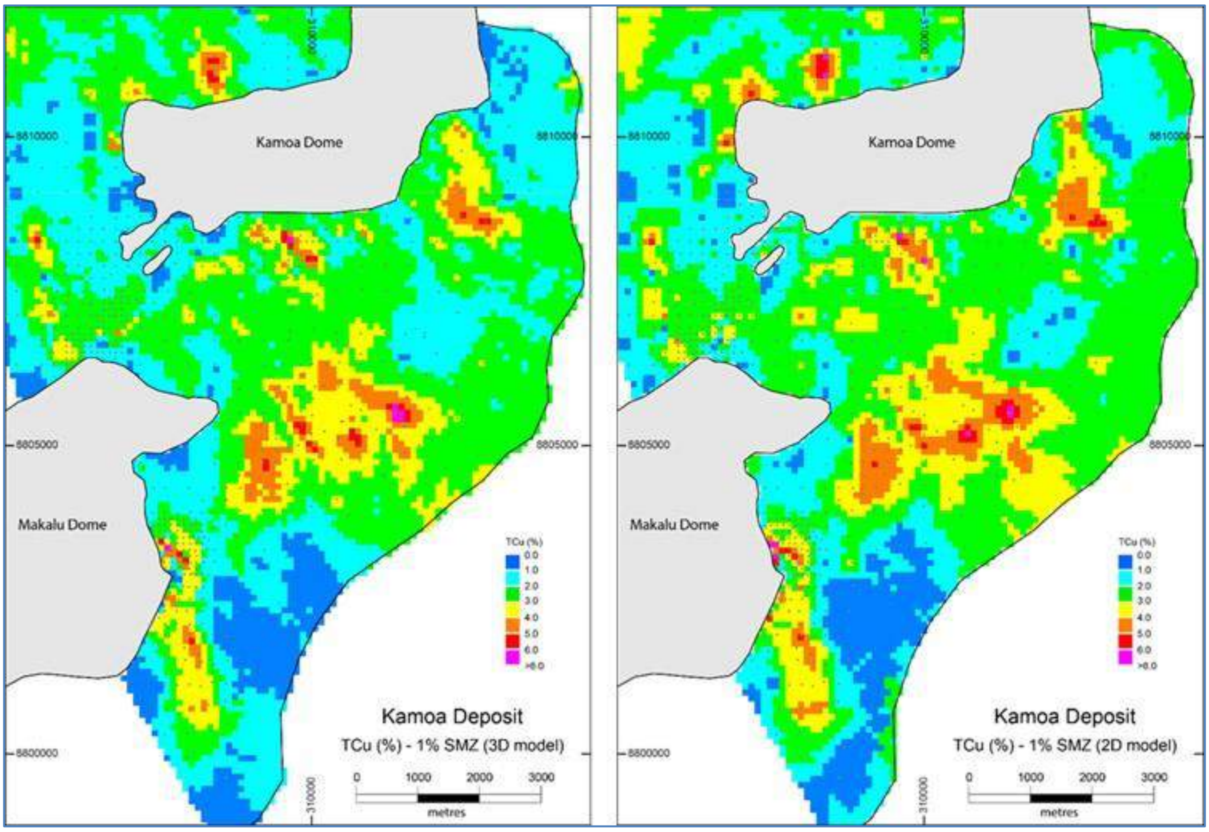


Figure prepared provided by Ivanhoe, 2017. Black dots are drillholes.

14.12 Global Bias Checks

14.12.1 Global Bias

During estimation, a declustered drillhole or NN estimate was included to allow a check for global bias between the estimated grade and the drillhole grades. The NN estimate minimises the effect of clustering of data and allows for a more appropriate comparison with estimated grades. This is particularly relevant at Kamoa, where a significant clustering of data occurs in the shallower portions of the deposit, close to the dome edges. Relative differences between the ID and NN models are generally below 5%, which is considered appropriate for an Indicated classification. Global biases are summarised in Table 14.13 for Kamoa, and in Table 14.14 for Kakula.

Table 14.13 Kamoa: Mean Grades for 1.0% Cut-off Composites and Models

Indicated (no cut-off applied)	Composite	Model (OK)	Model (NN)	Relative Diff (OK-NN)/NN
TCu (%)	2.56	2.35	2.28	3.1%
S (%)	1.60	1.66	1.69	-1.8%

Table 14.14 Kakula: Mean Grades for Domain 500 Composites and Models

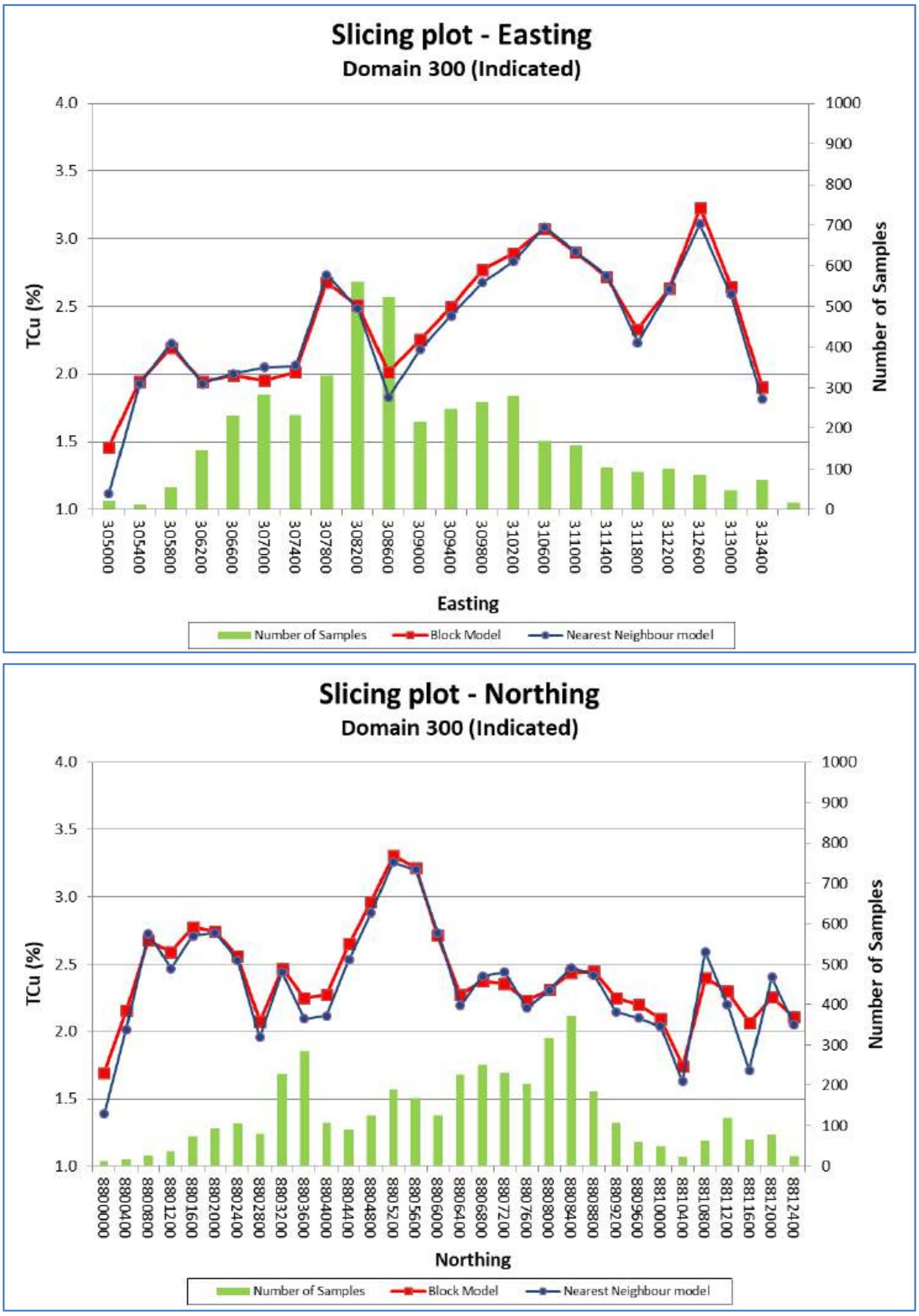
Indicated (No cut-off applied)	Composite	Model (OK)	Model (NN)	Relative Diff. (OK-NN)/NN
TCu (%)	4.78	4.42	4.43	-0.1%
S (%)	1.41	1.33	1.34	-0.4%

14.12.2 Local Bias Checks (Swath or Slicing Plots)

14.12.2.1 Kamoa

Checks for local bias were performed for TCu by analysing local grade trends on 400 m slices (swaths) in easting and northing. Example swath plots for TCu (%) are shown in Figure 14.23. The average grade per swath for the block model (red line) is compared with the average grade for the declustered drillholes (represented by the NN estimate) (blue line) for the same swath. The two lines are observed to follow very similar trends, indicating that no local biases are evident.

Figure 14.23 Kamoa: Swath Plots for TCu (%) for the Upper SMZ (Domain 300)



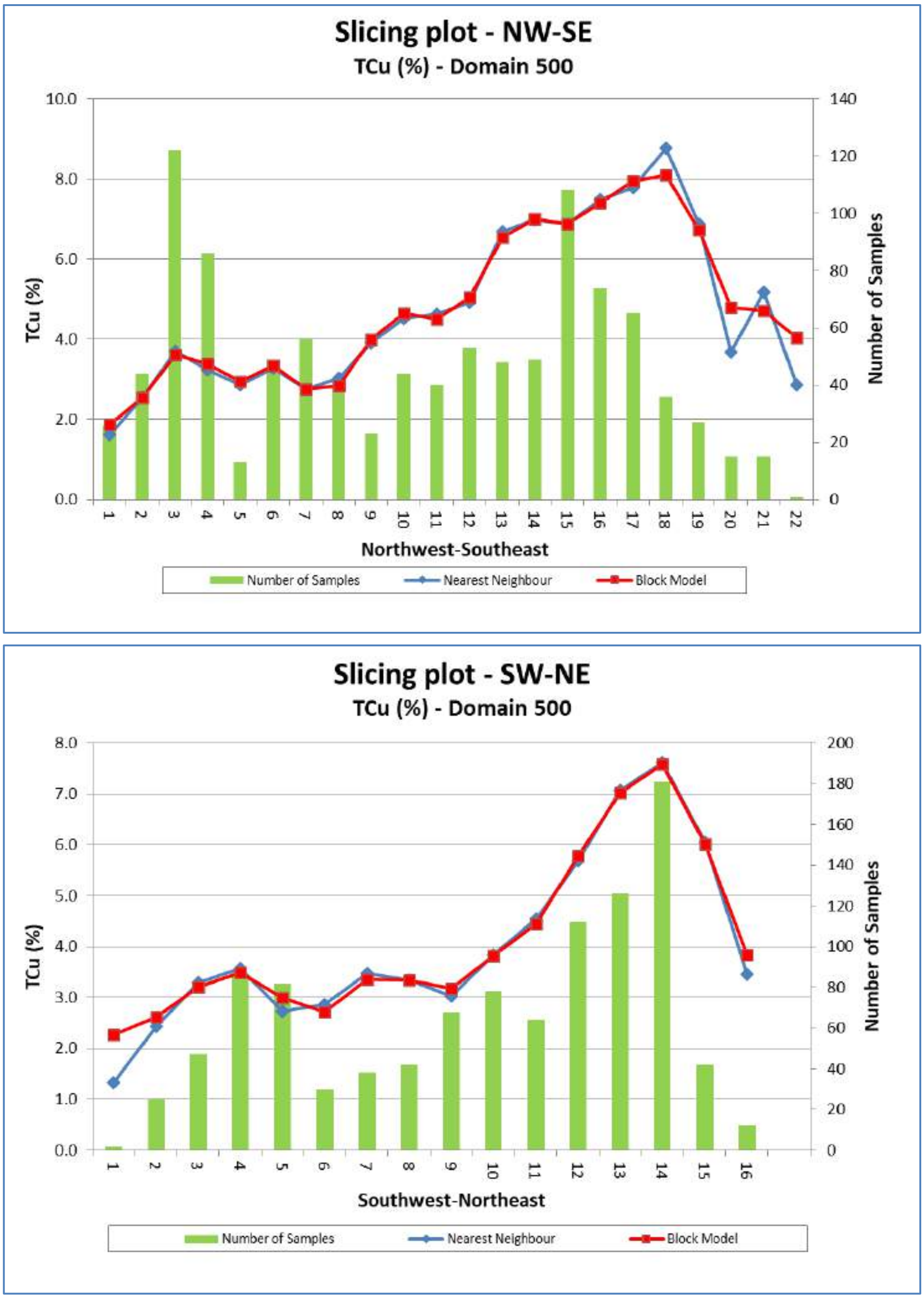
Figures prepared by Ivanhoe, 2018.

14.12.2.2 Kakula

Checks for local bias were performed for TCu and S by analysing local grade trends on 500 m swaths aligned north-west–south-east (along the trend of the high-grade mineralisation), and 500 m swaths aligned south-west–north-east (across the trend of the high-grade mineralisation).

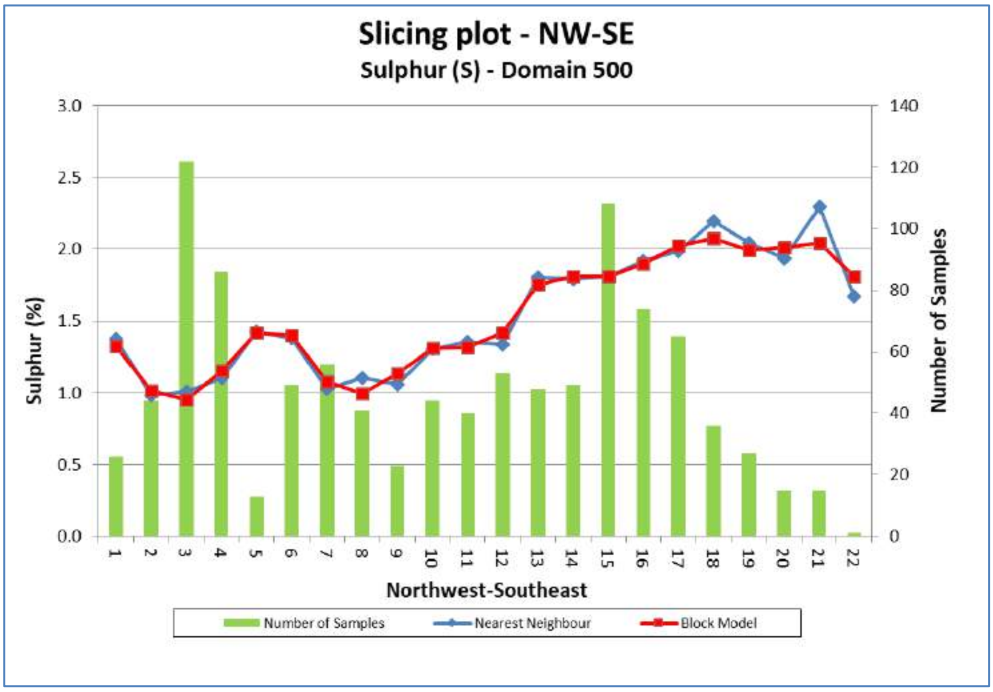
Example swath plots for TCu (%) and S (%) are shown for Domain 500 in Figure 14.24 and Figure 14.25, respectively. The average grade per swath for the block model (red line) is compared with the average grade for the declustered drillholes (represented by the NN estimate) (blue line) for the same swath. The two lines are observed to follow very similar trends, indicating that no local biases are evident.

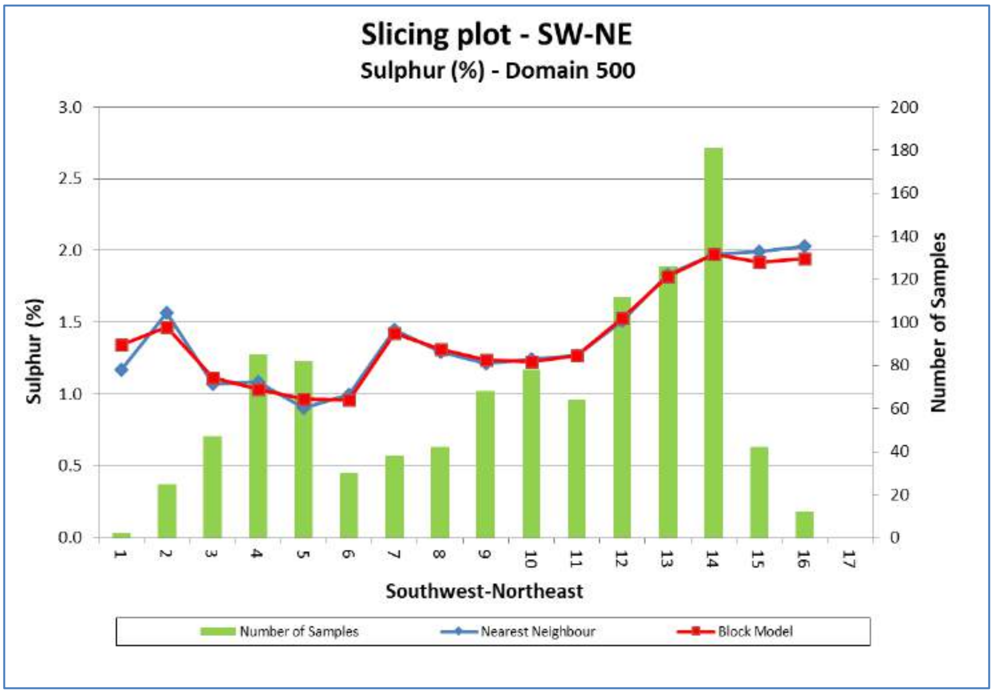
Figure 14.24 Kakula: Swath Plots for TCu (%) for the Basal Mineralised Siltstone (Domain 500)



Figures prepared and provided by Ivanhoe, 2019.

Figure 14.25 Kakula: Swath Plots for S (%) for the Basal Mineralised Siltstone (Domain 500)





Figures prepared and provided by Ivanhoe, 2019.

14.13 Reasonable Prospects of Eventual Economic Extraction

Amec Foster Wheeler has used a 1% TCu cut-off grade to support Mineral Resource estimation. This choice of cut-off is based on many years of mining experience on the Zambian Copperbelt at mines such as Konkola, Nchanga, Nkana, and Luanshya, which mine similar mineralisation to that identified at Kamoa and Kakula.

14.13.1 Kamoa Assessment of Reasonable Prospects for Eventual Economic Extraction

To test the cut-off grade for the purposes of assessing reasonable prospects of eventual economic extraction, Amec Foster Wheeler performed a conceptual analysis based on conditions considered appropriate for the region. A copper price of US$3.00/lb was assumed. The following additional key parameters were used:

- Percent recovery for hypogene is based on a reference case having a feed grade of 3.54% Cu and a tailings grade of 0.44% Cu. The reference case gives a recovery of 88.7%, and the adjusted recovery is 77.3% at a feed grade of 1.0% Cu.
- Percent recovery for supergene is based on a reference case having a feed grade of 3.54% Cu, and a tailings grade of 0.43% Cu. The reference case gives a metallurgical recovery of 88.7%. If the ASCu/TCu ratio is ≤ 0.125, the block is treated using hypogene recovery equations, but with an assumed concentrate grade of 45% TCu. If the ASCu/TCU ratio is > 0.125, the sulphide copper is estimated to be (TCu-ASCu) + 0.125 TCu. The hypogene recovery equation is then applied to sulphide copper, but with an assumed concentrate grade of 45% TCu. Estimated TCu recovery at a 1% TCu feed grade is 77.5% where the ASCu/TCu ratio is ≤0.125. Estimated TCu recovery at a 1% TCu feed grade where the ASCu/TCU ratio is 0.30 is 74.2%.
- Concentrate grades for supergene of 45.0% TCu and 16.9% S.
- Concentrate grades for hypogene of 36.0% TCu and 31.6% S.
- Concentrate moisture of 12%.
- Mining costs of US$27/t.
- Concentrator, tailings treatment and G&A costs of US$17/t treated.
- Payable copper of 97.1% for the supergene case and 96.4% for the hypogene case.
- Smelting costs of US$80/t of concentrates.
- Refining costs of US$0.08/lb payable copper.
- Transport costs of concentrates to smelter US$323/wmt concentrates.
- Royalty of 2% on payable copper – smelting costs – refining – transport costs.
- National Export Tax of 1% of payable copper – smelting costs – refining costs.
- Concentrate tax of US$100/wmt concentrates.
- NSR = payable copper – smelting costs – refining costs – transport costs – royalties – taxes.

The hypogene metallurgical recovery equation is based on the updated curve shown in Figure 13.14. The supergene metallurgical recovery equations are based on an analysis of the ASCu/TCu ratios of material designated as supergene:

- This analysis shows that 54% of the blocks classified as supergene have similar ASCu/TCu ratios as hypogene blocks. These blocks have ratios less than 0.125 and an average ratio of 0.084. These blocks (51% of all supergene blocks) are termed Group A.
- Where the ratio is > 0.125, their sulphide (floatable) copper is assumed to be (TCu-ASCu +0.125TCu). These blocks (49% of all supergene blocks) are termed Group B.

Group A blocks have metallurgical recoveries estimated using TCu and the hypogene equations. Group B blocks have recoveries estimated using sulphide copper as input to the hypogene equations. Figure 14.26 shows the % TCu recovery versus TCu feed grade. The overall TCu recovery for groups A+B is about 3.7% lower than would be obtained if the hypogene equation (Group A) were used throughout. The graph shows that the curve in Figure 13.14, based on Phase 6 ISF4a, is likely based on a non representative supergene sample with abundant copper oxides.

Figure 14.26 % TCu Recovery Versus % TCu in Feed for Kamoa Supergene Blocks

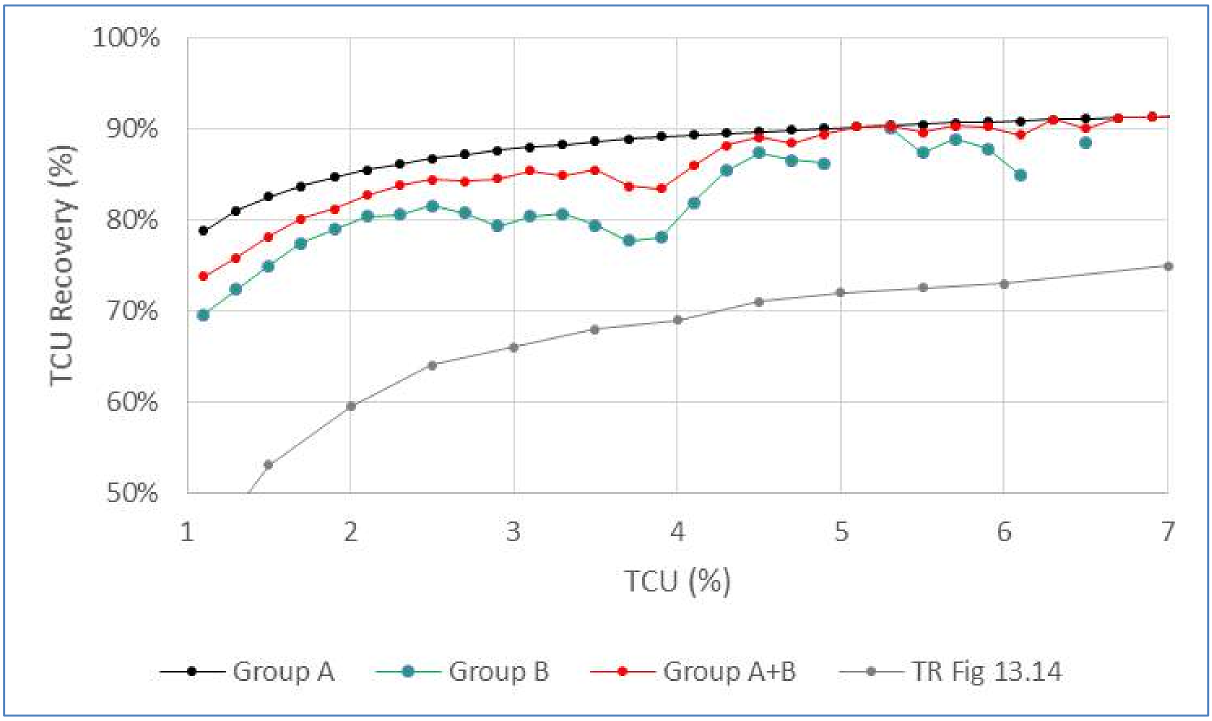


Figure provided by Amec Foster Wheeler, 2018.

Normally, cut-off grades used to declare Mineral Resources do not consider mining costs. There are additional areas for which reasonable prospects for eventual economic extraction exist and which might be scheduled if the nominal 12 Mtpa production rate used for the Kamoa-Kakula 2017 PEA was increased to as much as 20 Mtpa. These additional areas are included using a 1% TCu cut-off. There is a small percentage (~4%) of the tonnage representing 2% of the contained copper that has copper grades between 1.0% and 1.25%; the NSRs for these blocks will cover onsite concentrator, tailings treatment, and G&A costs but will not cover their full mining costs. Blocks grading between 1.0% and 1.25% TCu are estimated to cover \$23/t of \$27/t assumed mining costs. It may be convenient to mine these blocks in conjunction with adjacent higher-grade blocks, and therefore Amec Foster Wheeler has included the blocks in the Mineral Resource tabulations. Based on these assumptions, the Mineral Resources are considered to have met the requirement for reasonable prospects for eventual economic extraction.

As a sensitivity analysis, Amec Foster Wheeler considered a case in which an on-site smelter would produce blister copper (~99% Cu), as savings would be realised in terms of reduced transport of product costs. In addition, sulphuric acid of 98.5% purity would be produced for sale using a price of US\$200/t. This is perhaps a more realistic case in that the Kamoa resource base is large enough to contemplate on-site smelting (as was done for the 2013 PEA). For this case the NSRs for all blocks meeting a 1% Cu cut-off grade would cover onsite processing, tailings treatment, and G&A costs. There is a small percentage (0.4% of the tonnage and 0.2% of the metal) of blocks that will not cover full mining costs. These blocks will cover \$18/t of \$27/t assumed mining costs.

Amec Foster Wheeler cautions that with the underground mining methods envisioned (room-and-pillar or drift-and-fill), the mining recovery may vary from 55% to 80% depending on the success in which pillars can be mined on retreat, and/or a backfill or convergence method is used. In addition, the Mineral Resources do not incorporate allowances for contact (external) dilution at the roof and floor of the deposit. This will ultimately depend on the ability of the mining operation to follow the SMZ boundaries.

14.13.2 Kakula Assessment of Reasonable Prospects for Eventual Economic Extraction

In 2019 Amec Foster Wheeler assessed reasonable prospects for eventual economic extraction for Kakula (Peters et al., 2019). The assumptions incorporate a copper price of US$3.10/lb. It was found that the NSR (as defined below) for all Mineral Resources at a cut-off of 1% TCu will cover processing, tailings treatment, and general and administrative (G&A) costs. However, blocks grading between 1% and 1.3% TCu will cover most, but not all, of the full mining costs. It may be convenient to mine these blocks in conjunction with adjacent higher-grade blocks. These blocks represent 10% of the Mineral Resource tonnage and 5% of the contained copper. Based on this analysis, Amec Foster Wheeler considers the 2019 Mineral Resource estimates to be current for the purposes of this Technical Report. The assumptions made in Peters et al., (2019) broadly apply to Kakula and include an 18 Mtpa production rate and a concentrate grade of 57.3% TCu:

- Concentrator metallurgical recoveries range from 73% at a 1.0% TCu grade to 83% at the average grade of the Indicated Mineral Resource.
- Concentrate moisture of 8%.
- Mining costs of US$34.20/t.
- Concentrator, tailings treatment and G&A costs of US$20/t treated.
- Payable copper of 97.7%.
- Smelting costs of US$80/t of concentrates.
- Refining costs of US$0.08/lb copper in concentrates.
- Transport costs of concentrates to smelter US$253.8/t of Cu concentrates.
- Royalty of 3.5% on payable copper – smelting costs – refining costs – transport.
- National Export Tax of 1% of payable copper – smelting costs – refining costs.
- DRC tax on concentrates of US$100/mt.
- NSR = payable copper – smelting costs – refining costs – transport costs – royalties – taxes.

All mineralised material at Kakula is considered to be hypogene and is based on a reference case having a feed grade of 5.95% Cu and a tailings grade of 0.94% Cu. The reference case gives a metallurgical recovery of 86%, and the adjusted recovery is 73% at a feed grade of 1.0% Cu.

There are reasonable prospects for eventual economic extraction under assumptions of a copper price of US$3.10/lb, employment of underground mechanised room-and-pillar and drift-and-fill mining methods, and that copper concentrates will be produced. At a 1% TCu cut-off grade, the assumed NSRs for 100% of Mineral Resource blocks will cover concentrating, tailings treatment, and G&A costs.

As at Kamoa, there is a proportion (10%) of the tonnage representing only 5% of the contained copper in the Mineral Resource at Kakula that will not cover its full mining costs; e.g. blocks grading between 1% and 1.3% TCu. It may be convenient to mine these blocks in conjunction with adjacent higher-grade blocks, and therefore Amec Foster Wheeler has included the blocks in the Mineral Resource tabulations. For example, blocks grading between 1% and 1.3% TCu will have an average grade of 1.16% TCu, and these will cover $29/t out of the assumed $34/t mining costs. Based on these assumptions, the Mineral Resources are considered to have met the requirement for reasonable prospects for eventual economic extraction.

A similar analysis has been developed for an on-site smelting case. There are cost savings in shipping blister copper as opposed to concentrates. There is no export tax in this case and there are also acid credits of $250/t sulphuric acid produced. In this case blocks above 1.08% Cu will cover full mining costs as well as processing, tailings, G&A costs. These represent 98% of the tonnage and 99% of the metal in a resource grading above 1% TCu. Based on these assumptions, the Mineral Resources are considered to have met the requirement for reasonable prospects for eventual economic extraction if a smelter is built as part of the Project.

14.14 Mineral Resource Statement

The Mineral Resources were classified in accordance with the 2014 CIM Definition Standards. Mineral Resources are stated in terms of TCu, and an approximate minimum vertical thickness of 3 m.

To avoid reporting isolated blocks above cut-off in both the Kamoa and Kakula models, a minimum stack of three contiguous vertical blocks (3 m vertical thickness) was required to meet the cut-off criteria for the tonnage and grade estimate to be reported. In addition, where two or more distinct mineralised zones occurred in the same vertical profile, only the highest metal content zone was reported if the secondary mineralised zone could not justify the dilution between the two zones and remain above cut-off over the combined interval.

14.14.1 Kamoa Mineral Resource Statement

Indicated and Inferred Mineral Resources for the 3D resource model are summarised in Table 14.15. Mineral Resources are reported inclusive of Mineral Reserves on a 100% basis. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The Mineral Resources for Kamoa have an effective date of 27 November 2017. The Mineral Resources do not include any material in the hangingwall and footwall, and make no allowance for mining recovery factors.

Table 14.15 Kamoa Indicated and Inferred Mineral Resource (at 1% TCu Cut-off Grade)

Category	Tonnage (Mt)	Area (km²)	Copper (%)	Vertical Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
Indicated	759	50.7	2.57	5.5	19,500	43.0
Inferred	202	19.4	1.85	3.8	3,740	8.2

1. Ivanhoe's Mineral Resources Manager George Gilchrist, a Fellow of the Geology Society of South Africa and Professional Natural Scientist (Pr. Sci. Nat) with the South African Council for Natural Scientific Professions (SACNASP), estimated the Mineral Resources under the supervision of Dr. Harry Parker and Gordon Seibel, both Registered Members (RM) of the Society for Mining, Metallurgy and Exploration (SME), employees of Amec Foster Wheeler, who are the Qualified Persons for the Mineral Resource estimate. The effective date of the estimate is 27 November 2017 and the cut-off date for drill data is 23 November 2015. Mineral Resources are estimated using the CIM 2014 Definition Standards for Mineral Resources and Mineral Reserves. Mineral Resources are reported inclusive of Mineral Reserves on a 100% basis. Ivanhoe holds an indirect 39.6% interest in the Project. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
2. Mineral Resources are reported using a total copper (TCu) cut-off grade of 1% TCu and a minimum vertical thickness of 3 m. There are reasonable prospects for eventual economic extraction under assumptions of a copper price of US$3.00/lb, employment of underground mechanised room-and-pillar and drift-and-fill mining methods, and that copper concentrates will be produced and sold to a smelter. Mining costs are assumed to be US$27/t. Concentrator, tailings treatment, and general and administrative costs (G&A) are assumed to be US$17/t. Metallurgical recoveries are expected to average 84% (86% for hypogene and 81% for supergene). At a 1% TCu cut-off grade, assumed net smelter returns for 100% of Mineral Resource blocks will cover processing, tailings treatment and G&A costs.
3. Reported Mineral Resources contain no allowances for hangingwall or footwall contact boundary loss and dilution. No mining recovery has been applied.
4. Depth of mineralisation below the surface ranges from 10 m to 1,320 m for Indicated Mineral Resources and 20 m to 1,560 m for Inferred Mineral Resources.
5. Approximate drillhole spacings are 800 m for Inferred Mineral Resources and 400 m for Indicated Mineral Resources.
6. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.

14.14.2 Kakula Mineral Resource Statement

The Mineral Resources were classified in accordance with the 2014 CIM Definition Standards. Mineral Resources are stated in terms of TCu, and an approximate minimum vertical thickness of 3 m.

The Kakula Mineral Resource is a combination of separate Kakula and Kakula West resource models, with the West Scarp Fault forming the boundary between the two. The effective date of the estimate for Kakula is 13 April 2018, and the cut-off date for the drill data is 26 January 2018. The effective date of the estimate for Kakula West is 10 November 2018, and the cut-off date for the drill data is 1 November 2018. The Kakula Mineral Resource is summarised in Table 14.16 on a 100% basis. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The Mineral Resources do not include any material in the hangingwall and footwall dilution skins and make no allowance for mining recovery factors. All units are reported using metric units, with the exception of contained copper pounds, which is reported in imperial units.

Table 14.16 Combined Kakula: Indicated and Inferred Mineral Resource (at 1% TCu Cut-off Grade)

Category	Tonnage (Mt)	Area (km²)	Copper (%)	Vertical Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
Indicated	628	21.5	2.72	10.5	17,100	37.6
Inferred	114	5.9	1.59	6.9	1,810	4.0

1. Ivanhoe's Mineral Resources Manager George Gilchrist, a Fellow of the Geology Society of South Africa and Professional Natural Scientist (Pr. Sci. Nat) with the South African Council for Natural Scientific Professions (SACNASP), estimated the Mineral Resources under the supervision of Dr. Harry Parker and Gordon Seibel, both Registered Members (RM) of the Society for Mining, Metallurgy and Exploration (SME), employees of Amec Foster Wheeler, who are the Qualified Persons for the Mineral Resources. The Kakula Mineral Resource is a combination of separate Kakula and Kakula West models, with the West Scarp Fault forming the boundary between the two. The effective date of the estimate for Kakula is 13 April 2018, and the cut-off date for the drill data is 26 January 2018. The effective date of the estimate for Kakula West is 10 November 2018, and the cut-off date for the drill data is 1 November 2018. Mineral Resources are estimated using the CIM Definition Standards for Mineral Resources and Reserves (2014) and reported on a 100% basis. Mineral Resources are reported inclusive of Mineral Reserves on a 100% basis. Ivanhoe holds an indirect 39.6% interest in the Project . Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
2. Mineral Resources are reported using a total copper (TCu) cut-off grade of 1% TCu and a minimum vertical thickness of 3 m. There are reasonable prospects for eventual economic extraction under assumptions of a copper price of US$3.10/lb, employment of underground, mechanised, room-and-pillar and drift-and-fill mining methods, and that copper concentrates will be produced and sold to a smelter. Mining costs are assumed to be US$34/t. Concentrator, tailings treatment and general and administrative (G&A) costs are assumed to be US$20/t. Metallurgical recovery is assumed to average 83%. Ivanhoe is studying reducing mining costs using a controlled convergence room-and-pillar method. At a 1% TCu cut-off grade, assumed net smelter returns for 100% of Mineral Resource blocks will cover concentrator, tailings treatment and G&A costs.
3. Reported Mineral Resources contain no allowances for hangingwall or footwall contact boundary loss and dilution. No mining recovery has been applied.
4. Approximate drillhole spacings are 800 m for Inferred Mineral Resources and 400 m for Indicated Mineral Resources.
5. Rounding as required by reporting guidelines may result in apparent differences between tonnes, grade and contained metal content.

The Mineral Resource is divided into the Kakula and Kakula West areas in Table 14.17. The estimate presented in Table 14.17 is not additive to the estimate in Table 14.16.

Table 14.17 Kakula and Kakula West: Indicated and Inferred Mineral Resource split on the West Scarp Fault (at 1% TCu Cut-off Grade); Kakula – east of West Scarp Fault, Kakula West – west of West Scarp Fault

Category	Tonnage (Mt)	Area (km²)	Copper (%)	Vertical Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
Kakula						
Indicated	336	9.5	3.05	12.6	10,300	22.6
Inferred	33	1.9	1.45	6.2	473	1.0
Kakula West						
Indicated	292	12.0	2.33	8.8	6,810	15.0
Inferred	81	4.0	1.64	7.3	1,330	2.9
TOTAL						
Indicated	**628**	**21.5**	**2.72**	**10.5**	**17,100**	**37.6**
Inferred	**114**	**5.9**	**1.59**	**6.9**	**1,810**	**4.0**

The footnotes to Table 14.16 also apply to this table. The estimate presented in this table is not additive to the estimate in Table 14.16.

14.14.3 Kamoa–Kakula Project

Indicated and Inferred Mineral Resources for the Kamoa–Kakula Project are provided on a 100% basis in Table 14.18. The Mineral Resources in Table 14.15, Table 14.16 and Table 14.17 are not additive to this table.

Table 14.18 Kamoa and Combined Kakula: Indicated and Inferred Mineral Resource (at 1% TCu Cut-off Grade)

Deposit	Category	Tonnes (millions)	Area (Sq. km)	Copper Grade (%)	Vertical Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
Kamoa	Indicated	759	50.7	2.57	5.5	19,500	43.0
	Inferred	202	19.4	1.85	3.8	3,740	8.2
Kakula	Indicated	628	21.5	2.72	10.5	17,100	37.6
	Inferred	114	5.9	1.59	6.9	1,810	4.0
Total Kamoa-Kakula Project	Indicated	1,387	72.2	2.64	6.9	36,600	80.6
	Inferred	316	25.3	1.76	4.5	5,550	12.2

1. Ivanhoe's Mineral Resources Manager, George Gilchrist, Professional Natural Scientist (Pr. Sci. Nat) with the South African Council for Natural Scientific Professions (SACNASP), estimated the Mineral Resources under the supervision of Dr. Harry Parker and Gordon Seibel, both Registered Members (RM) of the Society for Mining, Metallurgy and Exploration (SME), who are the Qualified Persons for the Mineral Resource estimate. The effective date of the estimate for Kamoa is 27 November 2017, and the cut-off date for drill data is 23 November 2015.The Kakula Mineral Resource is a combination of separate Kakula and Kakula West models, with the West Scarp Fault forming the boundary between the two. The effective date of the estimate for Kakula is 13 April 2018, and the cut-off date for the drill data is 26 January 2018. The effective date of the estimate for Kakula West is 10 November 2018, and the cut-off date for the drill data is 1 November 2018. Mineral Resources are estimated using the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves. Mineral Resources are reported inclusive of Mineral Reserves, on a 100% basis. Ivanhoe holds an indirect 39.6% interest in the Project. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
2. Mineral Resources are reported for Kamoa using a total copper (TCu) cut-off grade of 1% TCu and a minimum vertical thickness of 3 m. There are reasonable prospects for eventual economic extraction under assumptions of a copper price of US$3.00/lb, employment of underground mechanised room-and-pillar and drift-and-fill mining methods, and that copper concentrates will be produced and sold to a smelter. Mining costs are assumed to be US$27/t. Concentrator, tailings treatment, and general and administrative (G&A) costs are assumed to be US$17/t. Metallurgical recovery will average 84% (86% for hypogene and 81% for supergene). At a 1% TCu cut-off grade, assumed net smelter returns for 100% of Mineral Resource blocks will cover concentrator, tailings treatment and G&A costs.
3. Mineral Resources are reported for Kakula using a TCu cut-off grade of 1% TCu and a minimum vertical thickness of 3 m. There are reasonable prospects for eventual economic extraction under assumptions of a copper price of US$3.10/lb, employment of underground, mechanised, room-and-pillar and drift-and-fill mining methods, and that copper concentrates will be produced and sold to a smelter. Mining costs are assumed to be US$34/t. Concentrator, tailings treatment and G&A costs are assumed to be US$20/t. Metallurgical recovery is assumed to average 83%. Ivanhoe is studying reducing mining costs using a controlled convergence room-and-pillar method. At a 1% TCu cut-off grade, assumed net smelter returns for 100% of Mineral Resource blocks will cover concentrator, tailings treatment and G&A costs.
4. Reported Mineral Resources contain no allowances for hangingwall or footwall contact boundary loss and dilution. No mining recovery has been applied.
5. Approximate drillhole spacings are 800 m for Inferred Mineral Resources and 400 m for Indicated Mineral Resources.
6. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.
7. The Mineral Resources in Table 14.15, Table 14.16, and Table 14.17 are not additive to this table.

14.15 Sensitivity of Mineral Resources to Cut-off Grade

Table 14.19 summarises the Kamoa Mineral Resource at a range of cut-off grades. The base case Mineral Resource model reported at a 1.0% TCu cut off is highlighted in grey. Mineral Resources reported in Table 14.15 and Table 14.18 are not additive to this table.

Table 14.19 Kamoa: Sensitivity of Mineral Resources to Cut-off Grade

<table>
<tr><th colspan="7">Indicated Mineral Resource</th></tr>
<tr><th>Cut-off (% Cu)</th><th>Tonnage (Mt)</th><th>Area (km²)</th><th>Copper (%)</th><th>Vertical Thickness (m)</th><th>Contained Copper (kt)</th><th>Contained Copper (billion lbs)</th></tr>
<tr><td>3.00</td><td>223</td><td>20.9</td><td>4.14</td><td>3.8</td><td>9,230</td><td>20.4</td></tr>
<tr><td>2.50</td><td>328</td><td>29.5</td><td>3.70</td><td>4.0</td><td>12,100</td><td>26.7</td></tr>
<tr><td>2.00</td><td>450</td><td>37.2</td><td>3.30</td><td>4.4</td><td>14,900</td><td>32.8</td></tr>
<tr><td>1.75</td><td>525</td><td>41.2</td><td>3.10</td><td>4.6</td><td>16,300</td><td>35.9</td></tr>
<tr><td>1.50</td><td>607</td><td>44.8</td><td>2.90</td><td>4.9</td><td>17,600</td><td>38.8</td></tr>
<tr><td>1.25</td><td>683</td><td>47.8</td><td>2.73</td><td>5.2</td><td>18,700</td><td>41.1</td></tr>
<tr><td>1.00</td><td>759</td><td>50.7</td><td>2.57</td><td>5.5</td><td>19,500</td><td>43.0</td></tr>
<tr><td>0.75</td><td>849</td><td>52.9</td><td>2.39</td><td>5.9</td><td>20,300</td><td>44.7</td></tr>
<tr><th colspan="7">Inferred Mineral Resource</th></tr>
<tr><td>3.00</td><td>19</td><td>2.1</td><td>3.52</td><td>3.2</td><td>655</td><td>1.4</td></tr>
<tr><td>2.50</td><td>40</td><td>4.4</td><td>3.10</td><td>3.3</td><td>1,250</td><td>2.8</td></tr>
<tr><td>2.00</td><td>67</td><td>7.1</td><td>2.77</td><td>3.4</td><td>1,840</td><td>4.1</td></tr>
<tr><td>1.75</td><td>88</td><td>9.2</td><td>2.55</td><td>3.5</td><td>2,240</td><td>4.9</td></tr>
<tr><td>1.50</td><td>114</td><td>11.8</td><td>2.34</td><td>3.5</td><td>2,660</td><td>5.9</td></tr>
<tr><td>1.25</td><td>151</td><td>15.3</td><td>2.10</td><td>3.6</td><td>3,170</td><td>7.0</td></tr>
<tr><td>1.00</td><td>202</td><td>19.4</td><td>1.85</td><td>3.8</td><td>3,740</td><td>8.2</td></tr>
<tr><td>0.75</td><td>253</td><td>22.5</td><td>1.65</td><td>4.1</td><td>4,180</td><td>9.2</td></tr>
</table>

The footnotes to Table 14.15 also apply to this table. Mineral Resources reported in Table 14.15 and Table 14.18 are not additive to this table.

Table 14.20 summarises the Kakula Mineral Resource at a range of cut-off grades. The base case Mineral Resource model reported at a 1.0% TCu cut-off is highlighted in grey.

Table 14.20 Combined Kakula: Sensitivity of Mineral Resources to Cut-off Grade

<table>
<tr><th colspan="7">Indicated Mineral Resource</th></tr>
<tr><th>Cut-off (% Cu)</th><th>Tonnage (Mt)</th><th>Area (km²)</th><th>Copper (%)</th><th>True Thickness (m)</th><th>Contained Copper (kt)</th><th>Contained Copper (billion lbs)</th></tr>
<tr><td>7.0</td><td>39</td><td>3.2</td><td>8.97</td><td>4.1</td><td>3,460</td><td>7.6</td></tr>
<tr><td>6.0</td><td>53</td><td>4.4</td><td>8.29</td><td>4.2</td><td>4,390</td><td>9.7</td></tr>
<tr><td>5.0</td><td>77</td><td>6.3</td><td>7.41</td><td>4.3</td><td>5,690</td><td>12.6</td></tr>
<tr><td>4.0</td><td>107</td><td>8.4</td><td>6.58</td><td>4.5</td><td>7,070</td><td>15.6</td></tr>
<tr><td>3.0</td><td>164</td><td>11.7</td><td>5.50</td><td>4.9</td><td>9,010</td><td>19.9</td></tr>
<tr><td>2.5</td><td>217</td><td>14.3</td><td>4.82</td><td>5.4</td><td>10,500</td><td>23.1</td></tr>
<tr><td>2.0</td><td>319</td><td>17.4</td><td>3.99</td><td>6.5</td><td>12,700</td><td>28.1</td></tr>
<tr><td>1.5</td><td>434</td><td>19.4</td><td>3.40</td><td>8.0</td><td>14,700</td><td>32.5</td></tr>
<tr><td>1.0</td><td>628</td><td>21.5</td><td>2.72</td><td>10.5</td><td>17,100</td><td>37.6</td></tr>
<tr><th colspan="7">Inferred Mineral Resource</th></tr>
<tr><td>4.0</td><td>1</td><td>0.1</td><td>4.40</td><td>3.4</td><td>32</td><td>0.1</td></tr>
<tr><td>3.0</td><td>5</td><td>0.4</td><td>3.52</td><td>3.9</td><td>163</td><td>0.4</td></tr>
<tr><td>2.5</td><td>11</td><td>1.0</td><td>3.09</td><td>3.7</td><td>325</td><td>0.7</td></tr>
<tr><td>2.0</td><td>23</td><td>2.1</td><td>2.62</td><td>3.9</td><td>604</td><td>1.3</td></tr>
<tr><td>1.5</td><td>47</td><td>3.9</td><td>2.16</td><td>4.3</td><td>1,010</td><td>2.2</td></tr>
<tr><td>1.0</td><td>114</td><td>5.9</td><td>1.59</td><td>6.9</td><td>1,810</td><td>4.0</td></tr>
</table>

The footnotes to Table 14.16 also apply to this table. This table is not additive to Table Table 14.16, Table 14.17, and Table 14.18

Table 14.21 summarises the Kamoa-Kakula Project Mineral Resource estimate at a range of cut-off grades. The base case Mineral Resource model reported at a 1.0% TCu cut-off is highlighted in grey.

Table 14.21 Kamoa and Combined Kakula: Sensitivity of Project Mineral Resources to Cut-off Grade

Indicated Mineral Resource						
Cut-off (% Cu)	**Tonnage (Mt)**	**Area (km²)**	**Copper (%)**	**Vertical Thickness (m)**	**Contained Copper (kt)**	**Contained Copper (billion lbs)**
3.0	387	32.7	4.72	4.2	18,200	40.2
2.5	545	43.8	4.14	4.4	22,600	49.8
2.0	770	54.7	3.59	5.0	27,600	60.9
1.5	1,040	64.3	3.11	5.8	32,400	71.3
1.0	1,390	72.2	2.64	6.9	36,600	80.6
Inferred Mineral Resource						
3.0	23	2.5	3.52	3.3	818	1.8
2.5	51	5.4	3.09	3.4	1,570	3.5
2.0	90	9.2	2.73	3.5	2,450	5.4
1.5	161	15.8	2.29	3.7	3,680	8.1
1.0	316	25.3	1.76	4.5	5,550	12.2

The footnotes to Table 14.18 also apply to this table. This table is not additive to Table 14.15, Table 14.16, Table 14.17, Table 14.18, Table 14.19, and Table 14.20.

14.16 Considerations for Mine Planning

The Kamoa deposit poses a significant challenge to building a reliable 3D model due to the deposit's lateral extent of tens of kilometres, and a vertical mineralisation extent of a few metres. These challenges, however, are minimised by the significant amount of high-quality drillhole data and the general consistency and predictability of the mineralisation.

Kamoa and Kakula were previously modelled using a 2D approach at a defined cut-off, or at a series of defined cut-offs. By averaging the grades over the full vertical extent of the SMZ, the vertical height of the mineralisation was fixed.

The 3D models provide the flexibility to locally vary the mining height to target narrower, higher-grade zones and locally adjust the vertical grade profile. This is especially useful in localised areas proximal to the growth faults in Kansoko Sud where the deposit was drilled at 50 m to 100 m grid spacing to account for the additional complexity. The 3D modelling method was designed to provide the flexibility to adjust the mining height or grade profile on a local scale to optimise the mine plan and improve the Project economics.

14.17 Targets for Further Exploration

Amec Foster Wheeler has identified a target for further exploration at Kamoa. It is referred to in this subsection as the Kamoa–Makalu exploration target. No targets have been defined for the Kakula deposit as the limits of the mineralisation have not been established.

The area inside the model perimeter surrounding the Indicated and Inferred Mineral Resources is shown in Figure 14.27. The ranges of the Kamoa–Makalu exploration target tonnages and grades are summarised in Table 14.22. Tonnages and grades were estimated using SMZ10 composites in the target area and applying a ±20% variance to the tonnages and grades.

Amec Foster Wheeler cautions that the potential quantity and grade of the Kamoa–Makalu exploration target is conceptual in nature, and that it is uncertain if additional drilling will result in the exploration target being delineated as a Mineral Resource.

Figure 14.27 Kamoa–Makalu Target for Further Exploration Location Plan

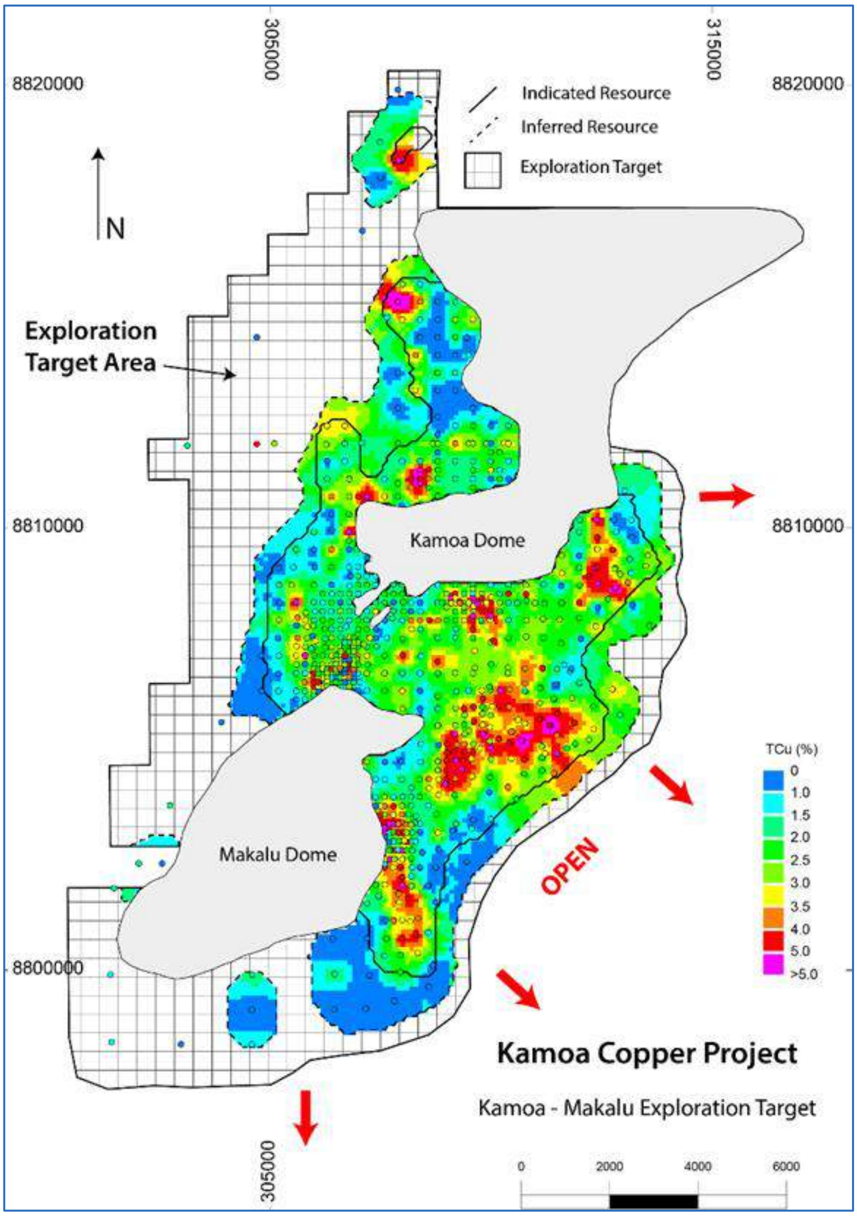


Figure by Ivanhoe, 2016. Scale bar represents metres.

Table 14.22 Kamoa–Makalu Target for Further Exploration: Tonnage and Grade Ranges

Target	Low-range Tonnage Mt	High-range Tonnage Mt	Low-range Grade (% Cu)	High-range Grade (% Cu)
Total	480	720	1.5	2.3

Additional Exploration Potential

The eastern boundary of the Mineral Resources at Kamoa is defined solely by the current limit of drilling, at depths ranging from 600 m to 1,560 m along a strike length of 10 km. Some of the best grade-widths of mineralisation occur here, and in addition, high-grade bornite-dominant mineralisation is common. Beyond these drillholes the mineralisation and the deposit are untested and open to expansion, even beyond the Kamoa-Makalu target for further exploration.

At Kakula, the western and south-eastern boundaries of the high-grade trend within the Mineral Resources are defined solely by the current limit of drilling. Exploration drilling is ongoing in these areas, and there is excellent potential for discovery of additional mineralisation.

14.18 Comments on Section 14

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). Amec Foster Wheeler has checked the data used to construct the resource model, the methodology used to construct them (Datamine macros) and has validated the resource model. Amec Foster Wheeler finds the Kamoa resource model to be suitable to support prefeasibility level mine planning, and the Kakula resource model is suitable to support a preliminary economic assessment.

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

- Drill spacing.
  - The drill spacing at the Kamoa and Kakula deposits is insufficient to determine the effects of local faulting on lithology and grade continuity assumptions. Local faulting could disrupt the productivity of a highly-mechanised operation. In addition, the amount of contact dilution related to local undulations in the SMZ has yet to be determined for both deposits. Ivanhoe plans to study these risks with the declines currently in progress at Kamoa and Kakula.
  - Delineation drill programs at the Kamoa deposit will have to use a tight (approximately 50 m) spacing to define the boundaries of mosaic pieces (areas of similar stratigraphic position of SMZs) in order that mine planning can identify and deal with these discontinuities. Mineralisation at Kakula appears to be more continuous compared to Kamoa.

- Assumptions used to generate the data for consideration of reasonable prospects of eventual economic extraction for the Kamoa deposit.
  - Mining recovery could be lower and dilution increased where the dip locally increases on the flanks of the domes. The exploration decline should provide an appropriate trial of the conceptual room-and-pillar mining method on the Kamoa deposit in terms of costs, dilution, and mining recovery. The decline will also provide access to data and metallurgical samples at a bulk scale that cannot be collected at the scale of a drill sample.
- Assumptions used to generate the data for consideration of reasonable prospects of eventual economic extraction for the Kakula deposit.
  - A controlled convergence room-and-pillar technique is being studied which provides the opportunity for reduced costs.
- Metallurgical recovery assumptions at Kamoa.
  - Metallurgical testwork at the Kamoa deposit indicates the need for multiple grinding and flotation steps. Variability testwork has been conducted on only portions of the Kamoa deposit. Additional variability testing is needed to build models relating copper mineralogy to concentrate grade and improve the recovery modelling.
  - A basic model predicting copper recovery from certain supergene mineralisation types has been developed. More variability testing is required to improve this model to the point where it is useful for production planning purposes.
- Metallurgical recovery assumptions at Kakula.
  - Preliminary metallurgical testwork at the Kakula deposit indicates that a high-grade chalcocite-dominant concentrate could be produced at similar or higher recoveries compared to those achieved for Kamoa samples.
  - There is no supergene mineralisation currently identified at Kakula that requires a dedicated recovery model separate from the hypogene recovery prediction method.
- Exploitation of the Kamoa-Kakula Project requires building a greenfields project with attendant 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 Kamoa and Kakula Mineral Resource estimates.
- Commodity prices and exchange rates.
- Cut-off grades.

15 MINERAL RESERVE ESTIMATES

15.1 Kamoa-Kakula Project Mineral Reserve

The Kamoa-Kakula Project Mineral Reserve includes the ore from both the Kakula Mine on the Kakula Deposit and Kansoko Mine at the Kamoa Deposit. The tonnes and grades were calculated for the mining blocks, and allowances for unplanned dilution and mining recovery were applied to calculate the Mineral Reserve Statement. The Total Probable Mineral Reserves are summarised in Table 15.1.

Table 15.1 Kamoa-Kakula Project Mineral 2019 Mineral Reserve

Classification	Ore (Mt)	Copper (%)	Copper (Contained Mlb)	Copper (Contained kt)
Proven Mineral Reserve	–	–	–	–
Probable Mineral Reserve	245.0	4.63	25,000	11,340
Mineral Reserve	245.0	4.63	25,000	11,340

1. Effective date of the all Mineral Reserves is 1 February 2019
2. Mineral Reserves are the total for the Kakula and Kansoko Mines.
3. The copper price used for calculating the financial analysis is long-term copper at US$3.10/lb. The analysis has been calculated with assumptions for smelter refining and treatment charges, deductions and payment terms, concentrate transport, metallurgical recoveries, and royalties.
4. For mine planning, the copper price used to calculate block model NSRs was US$3.00/lb.
5. An elevated cut off of US$100.00/t NSR was used to define the stoping panels. A marginal cut off of US$80.00/t NSR was used to define ore and waste.
6. Indicated Mineral Resources were used to estimate Probable Mineral Reserves.
7. Tonnage and grade estimates include dilution and recovery allowances.
8. The Mineral Reserves reported above are not additive to the Mineral Resources.

15.2 Kakula Mineral Reserve Estimate

The Kakula 2019 PFS Mineral Reserve has been estimated by Qualified Person Jon Treen, Senior Vice President, Stantec Consulting International LLC, using the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves to conform to the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects. The total Mineral Reserve for the Kakula Project is shown in Table 15.2. The Mineral Reserve is based on the 2017 Mineral Resource. The mineral reserve estimate in the 6 Mtpa scenario is based on the resource block model provided to Stantec in April 2018 (file name: mo_kakula3d_180413). Only the Indicated portion of the resource was used in estimating the mineral reserve. The Mineral Reserve is entirely a Probable Mineral Reserve that was converted from Indicated Mineral Resources. The effective date of the Mineral Reserve statement is 1 February 2019.

The Mineral Reserve defined in the Kakula 2019 PFS has not used all the Mineral Resources available to be converted to Mineral Reserve. The mining reserve focused on maximizing the grade profile for a 6 Mtpa full production rate for 15 years, with emphasis on further maximizing grade early in the mine life. Using the modeled targeted resource, tonnes and grades were calculated for mining shapes and allowances for unplanned dilution and mining recovery have been applied to calculate the Probable Mineral Reserves.

Table 15.2 Kakula 2019 PFS Mineral Reserves

Classification	Ore (Mt)	Copper (%)	Copper (Contained Mlb)	Copper (Contained kt)
Proven Mineral Reserve	–	–	–	–
Probable Mineral Reserve	119.7	5.48	14,475	6,566
Mineral Reserve	119.7	5.48	14,475	6,566

1. Effective date of the Kakula Mineral Reserve is 1 February 2019.
2. The copper price used for calculating the financial analysis is long-term copper at US$3.10/lb. The analysis has been calculated with assumptions for smelter refining and treatment charges, deductions and payment terms, concentrate transport, metallurgical recoveries and royalties.
3. For mine planning, the copper price used to calculate block model net smelter returns (NSRs) was US$3.00/lb.
4. An elevated cut-off of US$100.00/t NSR was used to define the stoping panels. A marginal cut-off of US$80.00/t NSR was used to define ore and waste.
5. Indicated Mineral Resources were used to estimate Probable Mineral Reserves.
6. Tonnage and grade estimates include dilution and recovery allowances.
7. The Mineral Reserves reported above are not additive to the Mineral Resources.

The internal and external dilution tonnage and the associated model grades were calculated based on the mining block dip, thickness, mining method, first or second lift extraction, and pillar design. Dilution shells 1 m thick were created around the mining blocks and interrogated against the block model, to determine the dilution grade associated with each individual mining block. Mining recoveries are based on mining block dip, thickness, and pillar requirements for each mining method.

The Mineral Reserve will be impacted by changes in revenue, costs, and other parameters. The elevated cut-off grades used to define the Mineral Reserve are a buffer against increases in cost or reduction in grade or recovery. The methodology used to define the Mineral Reserve has resulted in the highest-grade mining zones being identified to be mined first; this means that if the parameters vary positively or negatively, then it is likely that the mine plan, including the order of mining, will not change significantly.

As the mining production period was arbitrarily defined as 17 years, it is likely that further studies will define additional Mineral Reserves. This is supported by the large Mineral Resource that has already been defined.

Power supply to the project and continuity of supply are important factors that can affect the Mineral Reserve. To reduce the risk to the project, capital has been included for the power station upgrade to secure power for the project. This also allows more detailed studies to be undertaken to optimise the Kakula production capacity.

In the economic analysis, it has been assumed that rail will be available after two years and that there is therefore a significant reduction in concentrate transport costs, relative to the road transport assumption. This also provides a buffer against a reduction in Mineral Reserve.

15.3 Kamoa Mineral Reserve Estimate

This section has been updated to reflect a change in metal values and a total of 2% escalation to the costs from the Kamoa 2017 Development Plan.

The Kamoa 2019 PFS Mineral Reserve has been estimated by Qualified Person Jon Treen, Senior Vice President, Stantec Consulting International LLC, using the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves to conform to the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects. The total Mineral Reserve for the Kamoa Project is shown in Table 15.3. The Mineral Reserve is based on the 2017 Mineral Resource. The Mineral Reserve is entirely a Probable Mineral Reserve that was converted from Indicated Mineral Resources. The effective date of the Mineral Reserve statement is 1 February 2019.

The Mineral Reserve defined in the Kamoa 2019 PFS has not used all the Mineral Resources available to be converted to Mineral Reserve, as the analysis was constrained to produce a production period of 26 years. The Mineral Reserve is entirely contained within the Kamoa Mineral Resource. The two main areas within the Kamoa Mineral resource are: the Kansoko Sud and Centrale areas.

Table 15.3 Kamoa 2019 PFS Mineral Reserve

Classification	Tonnage (Mt)	Copper (%)	Contained Copper in Ore (Mlb)	Contained Copper in Ore (kt)
Proven Mineral Reserve	–	–	–	–
Probable Mineral Reserve	125.2	3.81	10,525	4,774
Mineral Reserve	**125.2**	**3.81**	**10,525**	**4,774**

1. Effective date of the Mineral Reserve is 01 February 2019.
2. The copper price used for calculating the financial analysis is long-term copper at US$3.10. The analysis has been calculated with assumptions for smelter refining and treatment charges, deductions and payment terms, concentrate transport, metallurgical recoveries and royalties.
3. For mine planning, the copper price used to calculate block model net smelter returns (NSRs) was US$3.00/lb.
4. An elevated cut-off of US$100.00/t NSR was used to define the stoping panels. A cut-off of US$80.00/t NSR was used to define ore and waste for the mine plan.
5. Indicated Mineral Resources were used to estimate Probable Mineral Reserves.
6. Tonnage and grade estimates include dilution and recovery allowances.
7. The Mineral Reserves reported above are not additive to the Mineral Resources.

The Kamoa 2019 PFS Mineral Reserve ranges between depths of 60 m and 1,300 m below surface, and the average dip is approximately 17°. Given the favourable mining characteristics of the Kamoa Mineral Resource, it is considered amenable to large-scale, mechanised, room-and-pillar mining or controlled convergence room-and-pillar mining. The saleable product will be copper concentrate. The processing production rate is 6 Mtpa ore.

Room-and-pillar mining will be used for ore zones from 60 m to 150 m in depth and from 150 m to 250 m in depth selectively during the production ramp-up period. For ore zones below 150 m not mined room-and-pillar during the ramp-up, controlled convergence room-and-pillar is the mining method of choice. Dip, depth, and mining height will define pillar size and post-destructive recovery. No postmining backfill will be required with these two methods.

Dilution has been applied as waste skins at the top and bottom contacts and by the use of footwall wedges below the orebody. Dilution was determined based on the method and shape, and mining losses were estimated as 2% for development and 5% for pillar extraction, to account for unrecovered ore.

Separate recoveries were applied to the Supergene and Hypogene metallurgical ore types. Smelter terms, concentrate transport, and royalties were applied to calculate the block model NSR. The NSR used for the Mineral Reserve definition assumed that concentrate transport was by road.

An NSR cut-off of $100.00/t was used to define the stoping panels. An NSR cut-off of US$80.00/t NSR was used to define ore and waste for the mine plan. Both these cut-offs are elevated relative to the breakeven cut-off that can be calculated from the cost assumptions in the economic analysis of the Kamoa 2017 PFS. The process, G&A, and mining costs that equate to the breakeven cut-off grade are approximately $46/t ore.

The Mineral Reserve will be impacted by changes in revenue, costs, and other parameters. The elevated cut-off grades used to define the Mineral Reserve are a buffer against increases in cost or reduction in grade or recovery. The methodology used to define the Mineral Reserve has resulted in the highest-grade mining zones being identified to be mined first; this means that if the parameters vary positively or negatively, then it is likely that the mine plan, including the order of mining, will not change significantly.

As the mining production period was arbitrarily defined as 26 years, it is likely that further studies will define additional Mineral Reserves. This is supported by the large Mineral Resource that has already been defined.

Power supply to the project and continuity of supply are important factors that can affect the Mineral Reserve. To reduce the risk to the project, capital has been included for the power station upgrade to secure power for the project. This also allows more detailed studies to be undertaken to optimise the Kamoa production capacity.

In the economic analysis, it has been assumed that rail will be available after two years and that there is therefore a significant reduction in concentrate transport costs, relative to the road transport assumption. This also provides a buffer against a reduction in Mineral Reserve.

16 MINING METHODS

16.1 Geotechnical Investigation

This section contains a summary of the PFS-level mining geotechnical investigation and design conducted for Kakula in Section 16.1.1, the Kamoa 2017 PFS and shown in Section 16.1.2. The PEA-level geotechnical investigation carried out by Kamoa Copper SA for Kakula West is shown in subsection 24.4.1.

16.1.1 Kakula Geotechnical Investigation and Design

This section contains a summary of the PFS-level mining geotechnical investigation and design completed by SRK Consulting (South Africa) Pty Ltd (SRK) for the Kakula 2019 PFS. This summary follows below.

The geotechnical investigation was based on geotechnical drilling and logging completed by Ivanhoe mines over the Kakula project area, which was reviewed and interpreted by SRK. This was accompanied by a laboratory testing programme of selected core samples obtained from the geotechnical and geological drilling done at the site. The results of various testing programmes (SRK 2017 and Cuprum 2017) the results for the Kakula PEA 2016 were included in this PFS investigation.

Geotechnical designs for room-and-pillar method and the drift-and-fill mining method were carried out by SRK during the PFS program discussed above.

16.1.1.1 Geotechnical Database

Geotechnical drilling and logging specific to the Kakula area were conducted by Ivanhoe mines. SRK completed five site visits to Kamoa during 2016 to 2018 for the purposes of geotechnical and structural logging QA/QC and data quality control on the Kakula Project.

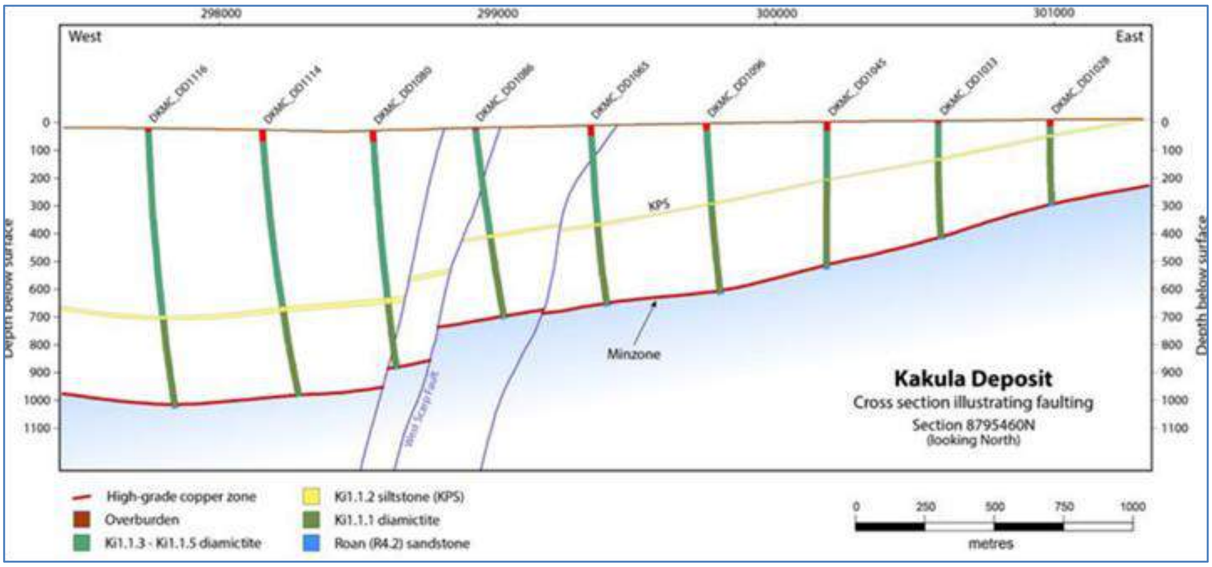
Findings from the visits have been documented in four letter reports (SRK 503457-2016, 505241-2016, 518054- 2017, 525297-2018) which outline on-site protocols, quality control reviews, details of the findings, recommendations for future data collection, and update aspects of various geotechnical and mining studies. Limited on-site data analysis and preliminary findings are also documented. Geotechnical data collection has improved over time. Recommendations have been made for regular follow-up visits as the project study level, data quantity, and required level of detail increases.

Geotechnical logs of 168 exploration drillholes were provided to SRK in Excel spreadsheets to form the basis of the rock mass classification for the Kakula 2019 PFS. The logging was carried out by a third-party consultant and the geotechnical logs were supplied to SRK. A detailed QA/QC exercise was conducted by SRK in the core yard between 14 and 16 August 2018 to verify the PFS geotechnical data. The logging was generally satisfactory, however some minor errors were observed in the recording of highly fractured zones. Adjustments were made to the database to improve the interpretation, where clear patterns emerged. Overall the SRK review showed that the geotechnical database was acceptable. SRK is confident that the database is suitable for this level of study.

Structural Domains

The 2018 geological model done by Ivanhoe indicates that there that three discrete structures which are north-north-east trending (see Figure 16.1) including and related to the West Scarp Fault (previously identified during the Kamoa PFS). These three structures have divided the mineral resource into four structural blocks. The planned extraction is to the east of these faults and these structures are not intersected by the planned mining effectively restricting the resource into one structural domain.

Figure 16.1 Kakula Section Looking North



The KPS siltstone is a stratigraphic layer above the orebody, which at Konsoko has a tendency to weather rapidly and the distance between the roof and this layer varies considerably and, in some areas, at Konsoko, it forms the hangingwall to the deposit. This however does not appear to be a problem at Kakula as can be seen in Figure 16.1 that the KPS is at least 300 m above the mineralised zone.

Intact Rock Properties

Geomechnical laboratory tests were undertaken for the three major lithologies diamictite(SDT), siltstone (SLT) that form the hangingwall and the orebody and sandstone (SST) that forms the footwall as part of the geotechnical investigation and these tests included:

- Uniaxial Compressive Strength (UCS) often with additional measurement of elastic properties (Young's modulus E and Poisson's ratio v);
- Brazilian Tensile Strength (BTS); and
- Triaxial Compressive Strength (TCS).

This data were used for rock mass classification and input for the numerical modelling;

Two accredited testing facilities were used, Rocklab (Scoping and PFS) in Pretoria South Africa and Cuprum (PFS) in Poland and both testing facilities conducted the tests according to the ISRM methodologies.

A summary of the rock properties measured in the laboratory for the three major lithologies is presented in Table 16.1. For elastic properties such as Poisson's ratio, some laboratory test results were excluded that is where the Poisson's ratio given was out of range. This explains why the count of Poisson's ratio is different from that of Young's Modulus. For TCS, the tests were binned according to three confining pressure ranges: 0 – 10 MPa, 10 – 20 MPa and 20 – 30 MPa.

Table 16.1 PFS Laboratory Testing Data (SRK 2017, SRK 2018 and Cuprum 2017)

Material Property	Stats	SDT	SSL	SST
UCS (MPa) (SRK)	**Number of tests**	24	1	15
	Min	6.5	8.3	39.0
	Average	88.3	8.3	159.0
	Max	142.2	8.3	332.8
	StdDev	32.8	0.0	94.5
UCS (MPa) (Cuprum)	**Number of tests**	61	5	17
	Min	8.2	9.0	11.0
	Average	63.6	36.4	59.5
	Max	136.5	100.1	160.7
	StdDev	29.4	33.7	40.9
E (GPa) (SRK)	**Number of tests**	24	1	15
	Min	25.7	7.3	44.6
	Average	66.6	7.3	70.3
	Max	77.7	7.3	79.3
	StdDev	12.3	0.0	10.2
E (GPa) (Cuprum)	**Number of tests**	61	5	17
	Min	3.3	4.8	2.8
	Average	16.1	8.7	16.1
	Max	33.5	17.7	33.8
	StdDev	7.1	4.8	8.0
Poisson's ratio v (SRK)	**Number of tests**	24	1	15
	Min	0.13	6.88	0.14
	Average	0.25	6.88	0.19

Material Property	Stats	SDT	SSL	SST
	Max	0.33	6.88	0.33
	StdDev	0.04	0.00	0.05
Poisson's ratio v (Cuprum)	**Number of tests**	41	2	14
	Min	0.01	0.05	0.03
	Average	0.16	0.06	0.14
	Max	0.46	0.08	0.36
	StdDev	0.10	0.02	0.10
BTS (MPa) (SRK)	**Number of tests**	50		23
	Min	2.9		6.3
	Average	9.3		13.8
	Max	17.9		23.4
	StdDev	3.0		4.8
BTS (MPa) (Cuprum)	**Number of tests**	35	13	13
	Min	1.7	1.7	3.1
	Average	7.8	5.2	8.2
	Max	15.2	11.5	12.1
	StdDev	2.9	2.8	2.8
TCS (MPa) (SRK)	**Number of tests**	45	3	4
	Min	77.9	119.7	84.5
	Average	187.8	225.0	157.4
	Max	391.5	323.3	282.2
	StdDev	62.7	83.3	78.3
BFA (°) (SRK)	**Number of tests**	18	2	13
	Min	28.0	32.0	28.0
	Average	34.3	34.5	34.3
	Max	39.0	37.0	38.0
	StdDev	2.6	2.5	2.7

The small sampling theory was used to quantify the variability of rock mass properties. This theory is based on statistical inferences and is suitable for situations where the number of data points is less than 40 as is often the case with laboratory tests due to cost constraints. The small sampling theory uses a student-t distribution. The small sampling theory assessment found the following:

- The current state of laboratory testing in the SDT is adequate to allow the use of this data for designing permanent excavations during the operational phase.
- The data in SST is adequate for the prefeasibility phase.

The data obtained for SSL data is suitable for the prefeasibility study. However for the feasibility study more laboratory tests are recommended in the feasibility study, approximately 62 UCM tests.

Rock Mass Classification

The rock mass properties at Kakula were inferred from geotechnical logging data from 168 drillholes and the laboratory testing programmes discussed above. The logging of the drillholes was carried out by mine personnel at the site. A geotechnical core logging system, originally developed by SRK, has been applied to capture the core recovery, RQD, rock hardness, fracture frequency and joint strength characteristics. The raw data is used to calculate both Laubscher's (1990) RMR and Barton's Q.

The Laubscher (1990) rock mass classification results were used for pillar design and Barton's Q system was used for the support design. The logging data were also used to identify potentially hazardous conditions such as weakness zones within the planned mining horizon. The ore zone/mining Horizon was defined by the 3% CU cut-off and the hangingwall and the footwall were taken to be 5 m above and 5 m below the ore zone respectively. Effectively the hangingwall consists of diamictite, similar to the ore zone whereas the footwall consists of sandstone.

Geotechnical Domains

The geotechnical characteristics are influenced by the three primary lithological domains, namely siltstone, diamictite and sandstone (from the Roan basement rock). The rock mass characteristics indicated following geotechnical zones:

- From the RMR values, the ground at Kakula belongs to the classes of poor and to a lesser extent, fair ground;
- The Q-values estimated that fair gedground conditions account for over 50% of the entire mine area, good ground for 7%, poor ground over 30% and very poor for 13%;
- It is important to note that the spacing in the PFS drilling grid is 200 m on average. With such a wide spacing, details are lost, and the contours presented in this study are often purely indicative. Additional drilling is required for the feasibility study near areas of poor rock mass conditions to delineate better the extent of these zones.

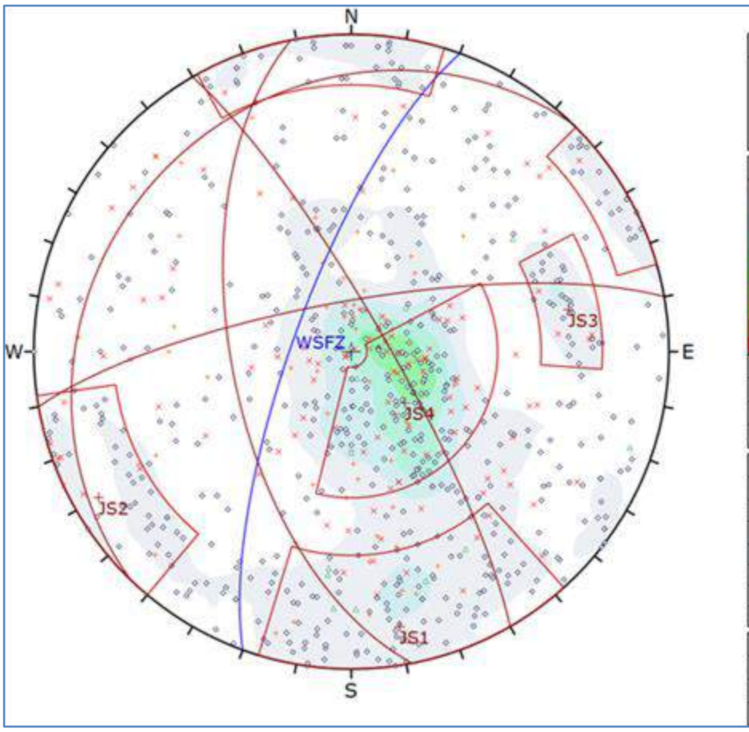
- The footwall stratigraphy was characteristised to approximately 10 m of the upper portion of the sandstone, this is where boreholes typically terminated. It was recognised that a potential footwall aquifer exists beneath, and the ground characteristics of aquifer could influence the stability of excavations in the footwall. Therefore, it is important that Kamoa describes the characteristics and the porosity of the sandstone for the feasibility study.

Structural Assessment

To ensure that the data that were used for the structural assessment was of a required standard, 14 boreholes were relogged by a third party and these data were then provided to SRK for a structural assessment to establish joint patterns across the orebody. Of the 14 boreholes, 12 were used for the structural assessment because 2 of the relogged holes did not contain beta values.

The structural assessment of the data was completed by SRK using the data provided and Figure 16.2 shows all the joint sets which are present in the project area.

Figure 16.2 Stereographic Projection for the Kakula 2019 PFS Project Area



Symbol	LITH	Quantity
◇	SDT	591
×	SSL	114
▲	SST	13
•	[no data]	56

Color	Density Concentrations
⬜	0.00 - 1.00
⬜	1.00 - 2.00
⬜	2.00 - 3.00
🟩	3.00 - 4.00
🟩	4.00 - 5.00
🟩	5.00 - 6.00
🟨	6.00 - 7.00
🟨	7.00 - 8.00
🟧	8.00 - 9.00
🟥	9.00 <

Contour Data	Pole Vectors
Maximum Density	4.83%
Contour Distribution	Fisher
Counting Circle Size	1.0%

	Color	Dip	Dip Direction	Label
		User Planes		
1	🟦	75	290	WSFZ
		Mean Set Planes		
1m	🟥	77	350	JS1
2m	🟥	81	60	JS2
3m	🟥	59	259	JS3
4m	🟥	19	314	JS4

Plot Mode	Pole Vectors
Vector Count	774 (774 Entries)
Hemisphere	Lower
Projection	Equal Area

The following was established during this structural assessment.

Joint Patterns

Four main joint sets were found to be present across the project area:

- JS1 is a major dominant set which is steeply dipping to the north.
- JS2 is a major joint set steeply dipping to the north-east.
- JS3 is a prevalent joint set which dips to the south west. This set may be somewhat related to the west scarp structures which dips steeply in a westerly direction (to the north-west).
- JS4 is a sub-horizontal discontinuity sets and is thus interpreted as a bedding.

Discontinuities are generally planar to undulating and, in most cases, contain either a fine non-softening or fine softening infill.

The joint sets present in the hangingwall are like those present in the footwall.

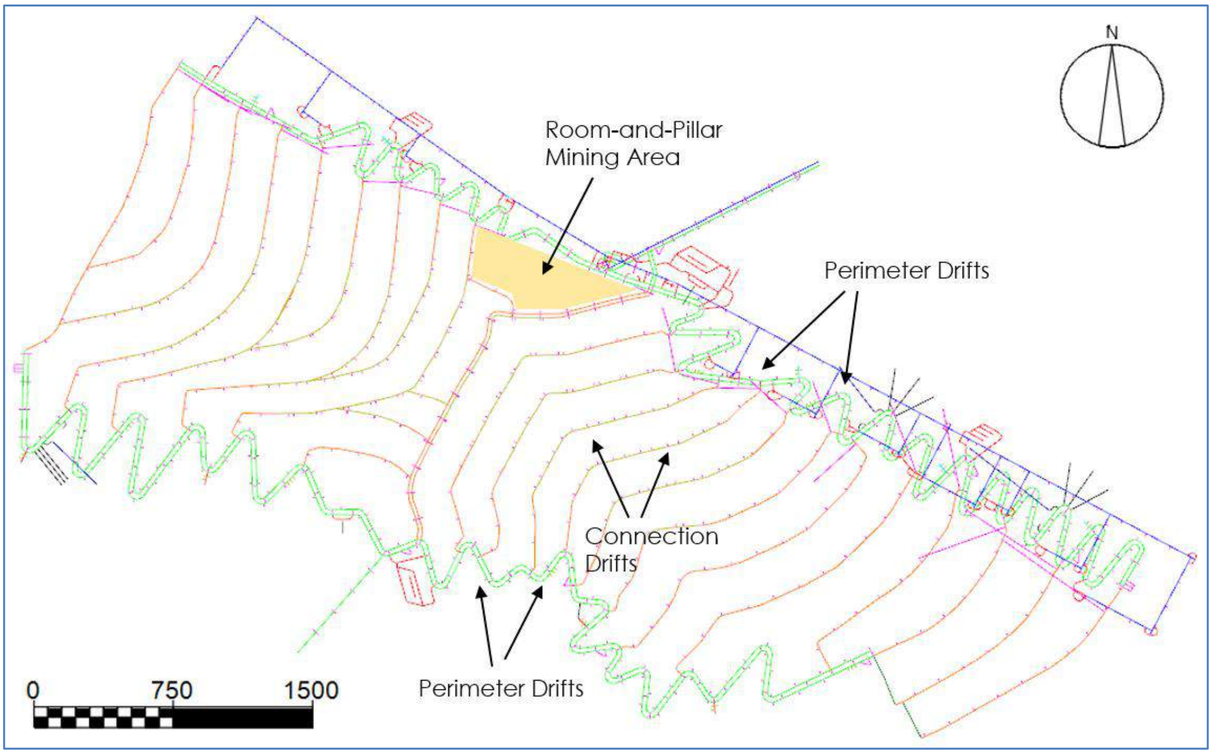
Using the data obtained the jointing trends in the orebody itself are difficult to establish. However, considering the predominant use of the cut and fill mining method it is considered that the structural assessment is adequate for a prefeasibility level of study.

There are currently seven structural boreholes and one ATV borehole across the PFS project area. It is important for the feasibility study that additional boreholes be logged either structurally and/or geophysically (ATV) to verify and improve confidence in the discontinuity sets that have been identified.

16.1.1.2 Geotechnical Design

The mining methods and layouts are described in Section 16.2. The two mining methods to be used are room-and-pillar and drift-and-fill. The room-and-pillar mining method is only planned for a small area adjacent to the infrastructure associated with the northern access to the orebody as shown in Figure 16.3.

Figure 16.3 Mine Infrastructure Illustrating Accessways and the Room-and-Pillar Mining Area



The room-and-pillar design provides for a stiff, non-yielding system in which excavations remain open for the LOM. This mining method significantly reduces the extraction ratios. The remainder of the orebody is to be extracted using the drift-and-fill mining method. This mining method will include a paste fill and as such little to no closure is expected to occur in the mined-out area.

Geotechnical Discussion of Mining Methods

Room-and-pillar (one cut with a maximum height of 6 m) will be used in a small portion of the orebody. There is abundant experience in the application of room-and-pillar mining to tabular orebodies in a wide range of geological environments. It is notable that large-scale room-and-pillar mining have been associated with unexpected massive collapses due to sudden failures over an extensive area. The key requirements for successful application of the room-and-pillar method is a proper understanding of the stability of the rooms and ensuring that the in-panel pillar layouts are adequate for the expected conditions. Taking this into account the room-and-pillar mining method has been adequately designed.

The drift-and-fill mining method will be used for the extraction of the majority of the Kakula and will consist of connection drifts with production drifts developed across the orebody from which the production drifts/headings extend on either side of the connection drifts. Based on the orebody thickness as defined by the 3% cut-off boundaries two lifts (benches) will be required in a relatively large portion of the orebody. The current analysis considers the first 6 m lift and some disturbance and changes to the rock mass that would take place in the hangingwall because of the first cut. However, if the paste fill is filled tightly this will significantly reduce unravelling in the hangingwall. It is imperative that the paste fill is tightly filled. Tight filling will also assist in maintaining the integrity of the connection drift protection pillars.

Access to the production area will be provided by the primary drifts where the secondary and tertiary drifts will be established from the primary production drifts to facilitate the establishment of protection pillars either side of the connection drifts. Each production drift will be paste filled and cured prior to the removal of the drift adjacent to the completed production drift. The objective of these protection pillars will assist in ensuring the long-term stability of the connection drifts which are required for access, ventilation and provision to the production drifts. These pillars will then be extracted on the retreated once the production headings have been filled with paste fill and cured. The paste fill and the production pillars have been adequately designed to facilitate the drift-and-fill mining method.

Pillar Design (Room-and-Pillar)

Pillar strength for the room-and-pillar mining method has been designed adequately to provide for a stiff, non-yielding system for the areas where this mining method is to be used.

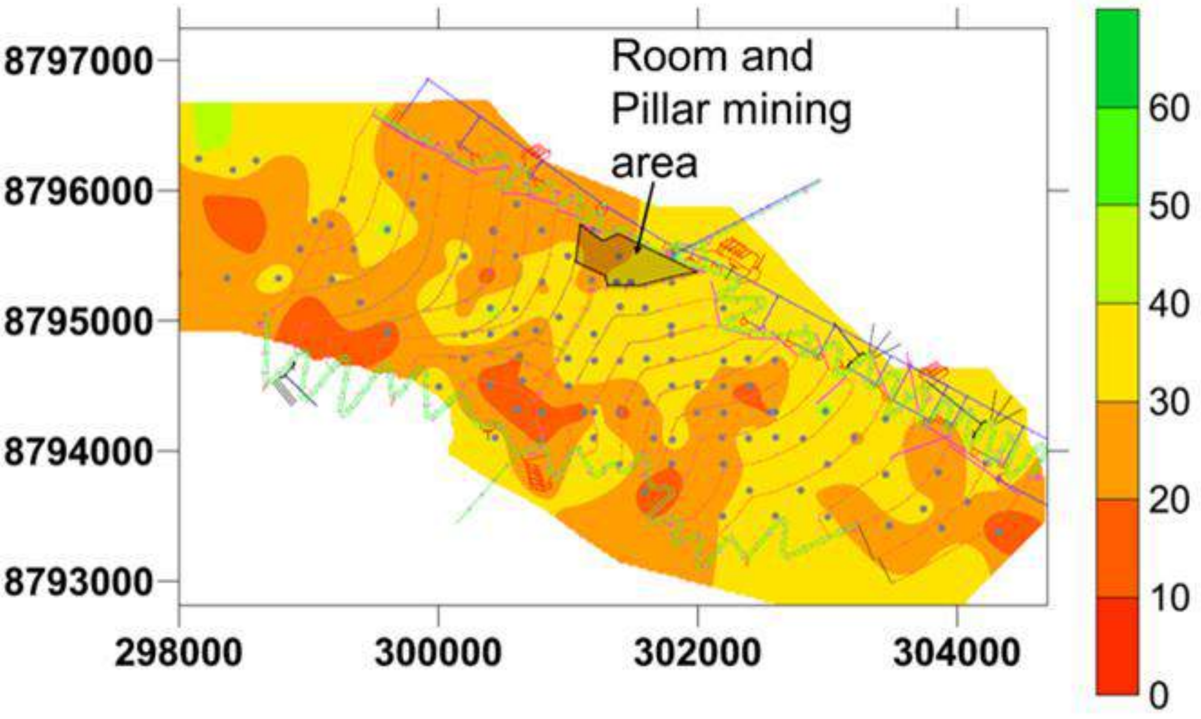
The pillar design and theoretical extraction ratios for a range of depth intervals and mining heights were based on Tributary Area Theory (TAT) for square and rectangular pillars. The Kamoa resource and surrounding rock conditions change significantly across the project area. A variety of pillar designs are provided to accommodate the changing rock mass conditions.

In-panel pillar designs are based on the Hedley and Grant (1972) empirical formula. The formula derives the in-situ pillar strength from the DRMS and the pillar dimensions. Stability pillar have not been included in the pillar design for the room-and-pillar mining method because the spans of the planned mining area for this method are not excessive.

The in-panel pillar loads were calculated using the Tributary Area Theory (TAT) that assumes that pillars carry the entire load to surface and this is shared equally by all the pillars. The pillar load is a function of the virgin vertical stress and extraction ratio.

The in-stope pillar for the room-and-pillar mining method was designed using the Hedley and Grant methodology with a Design Rock Mass Strength (DRMS) of 30 MPa. Pillar sizes were designed for range of depths and for varying dips. The contour plot (Figure 16.4) show that the DRMS ranges 20 and 30 MPa in the area where room-and-pillar mining is planned. This was assessed and the 2 boreholes in the lower DRMS contour indicate that the DRMS in that area is between 28 and 30 MPa. It must however be noted that a DRMS of 30 MPa is not conservative and the pillar strength may be reduced in the lower DRMS area requiring that the pillar sizes need to be increased to cater for this as the room-and-pillar mining moves toward the East.

Figure 16.4 Room-and-Pillar Mining with DRMS



Pillar and Fill Design for the Drift-and-Fill Mining Method

The paste fill was designed to provide confinement to the excavation surfaces and to prevent caving and roof fall in conjunction with the support that was designed for the connection drives and the production drifts. The paste fill was also designed to ensure that the fill was self-supporting during the second and third stages of extraction when the adjacent drifts are extracted. The paste fill requirements were determined using an analytical method for static analysis of backfill described by Mitchell (1981). The paste fill strength requirements are 90 kPa to obtain a free-standing height of 7.0 m. The paste fill will require to be supported with split sets and mesh to prevent sidewall slabbing due to loading during the mining cycle. A bulkhead will be required during the backfill placement and these bulkheads will require a 20 MPa strength with a thickness pf 0.4 m to contain the hydrostatic head that will be experienced in a 7 m high drift at the time of the backfill placement.

SRK has used the MAP3D numerical software to assess the pillar requirements that would ensure the long-term stability of the connection drifts. Map3D applies Boundary Element Method (BEM) of stress analysis and assumes that the rock is elastic, homogeneous and continuous. The input parameters for the numerical model were obtained the laboratory tests and the rock mass classification process discussed above. A failure criterion using a Hoek and Brown rock strength parameters was used to estimate the amount of damage that will result from the stress on the pillars that will occur during the mining cycle. Pillar size requirements for the protection pillars were determine for different dips and depths and are shown in Table 16.2.

Table 16.2 Connection Drift Protection Pillars Requirements

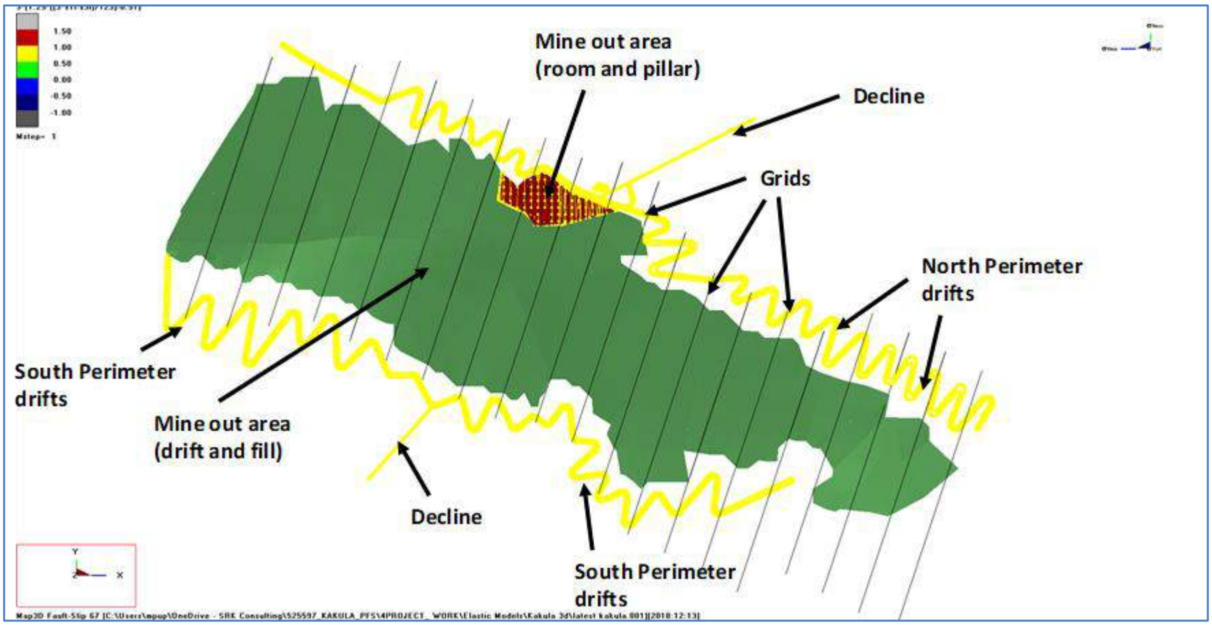
Depth (m)	Plan Pillar Length (m)					
	0°–12°	13°–16°	17°–20°	*21°–25°	*26°–30°	*31°–35
400	20	28	34	29	33	37
600	25	33	39	34	38	42
800	30	38	44	39	43	47
1000	35	43	49	44	48	52

*All sides are evaluated by the apparent dip.

Protection of Perimeter Drifts

A numerical assessment of the stability of the perimeter drifts when exposed to the planned mining in the Kakula orebody using the MAP3D numerical modelling software. The overall model geometry is shown in Figure 16.5.

Figure 16.5 Overall Model Geometry



This numerical exercise showed that the depth of failure in all cases would be less than 1.5 m and this depth of failure would be catered for with the support recommended for these areas.

It must be borne in mind that all models are a simplification of reality and consequently the actual response of the hangingwall, footwall and protection pillars need to be quantified during the mining process. This will require a suitable monitoring process which should be included in the feasibility study.

Support Design

Ground support is assigned based on mining method, location and ground condition.

The installation of the support all support 1.5 m above the footwall is of importance in the connection drifts due to the potential sidewall closure occurring lower in the connection drifts. This support is also required 1.5 m above the footwall of the pillars because of the potential of the bottom of the sidewall being forced in to the excavation by stress induced failure if this area is not supported.

Given that the diagonal distance at the intersections is large, intersection support has been included. Cable bolts have been recommended, but yielding bars at the intersections are not required in the room-and-pillar area.

Preliminary Subsidence Review

Surface subsidence above the drift-and-fill stopes will be negligible considering that the stope will be backfilled, and that mining depth is greater than 200 m below surface. Displacement of the hangingwall area will increase as the mining span increases but will be resisted by the backfill. Due to the contiguous placement of backfill in adjacent drifts, the backfill will be confined and will therefore increase resistance as the deformation increases. Stope hangingwall deformation is therefore expected to be purely elastic and in the order of centimetres. Surface deformation, 200 m above the stope will therefore be negligible. For the room-and-pillar mining area where the in-panel pillars are designed to carry the load to surface no subsidence is expected.

16.1.1.3 Conclusions and Potential Geotechnical Risks

The geotechnical data have been collected according to internationally acceptable standards and QA/QC reviews were done onsite to confirm compliance to data collection standards. Rock material testing of the main lithological has been done to establish typical rock mass strengths and elastic properties. Overall the work done is suitable for PFS requirements.

The geotechnical risks for this project were identified and are summarised below:

- The uncertainty due to the wide-spacing of data and lack of understanding of the frequency of structures and their deformation zones, that may impact the competency of the underground rock mass and the continuity of the deposit.
- The actual response of the hangingwall, footwall and in-panel pillars need to be quantified during the implementation stage to quantify the integrity of the in-stope pillars.
- The pillar design and extraction percentages are based on the summarised data obtained from drillhole core only. The information is considered representative but needs to be verified through data collection from underground exposures.
- The stability of the rock mass within the mining environment is not well known, specifically with respect to geological structures contained in the pillars and the hangingwall.
- It is important to retreat towards a stable access during pillar extraction to ensure safety of personnel.

16.1.1.4 Recommendations for Feasibility Study

The geology in the area includes significant geological structure with numerous faults and a wide range of joint orientations. The occurrence and the condition of these structures needs to be better understood. The behaviour of the hangingwall will be affected by geological structure and orientation. It is recommended that a full scale geotechnical mapping of the rock mass is done during the development and trial mining phase.

It is recommended that an additional 62 UCM tests for SSL are required for the feasibility study.

It is important to note that the spacing in the PFS drilling grid is 200 m on average. With such a wide spacing, details are lost, and the contours presented in this study are purely indicative. Additional drilling is required for the feasibility study near areas of poor rock mass conditions to delineate better the extent of these zones.

The footwall stratigraphy was characterised for approximately 10 m of the upper portion of the sandstone. It was recognised that a potential footwall aquifer exists below this, and the ground characteristics of aquifer could influence the stability of excavations in the footwall. Therefore, it is important that Kamoa describes the characteristics and the porosity of the sandstone for the feasibility study.

There are currently seven structural boreholes and one ATV borehole across the PFS project area. It is important for the feasibility study that additional boreholes be logged either structurally and/or geophysically (ATV) to verify and improve confidence in the discontinuity sets that have been identified.

The stress environment is unknown at this stage and a k ratio of 1 was assumed for the numerical modelling assessment. However, if the horizontal stress is significantly higher this could result in an increase in the depth of failure in the hangingwall of the long-term excavations. Similarly, for the drift-and-fill mining method a high horizontal stress may have a detrimental effect on the relaxation that will occur in the hangingwall of the first cut. Therefore, for the feasibility study It is recommended that initially the modelling is done with varying k ratios to determine the potential effect of high horizontal stress. In addition to this it is recommended that during the trial period stress measurement is done to establish the magnitude and direction of the virgin stress.

The layout and sequences will need to be optimised in the next phase of study to ensure safety of personnel and long term excavations.

Additional rock mass characterisation of boreholes is required to obtain better resolution across the orebody especially in the footwall of the orebody.

Additional footwall and triaxial tests are recommended for the orebody as well as for the footwall to reduce uncertainty in terms of rock mass strength and help to accurately define Hoek-Brown rock mass strength parameters. The selection of specimens for these tests should be carried out against well-defined boundaries of the orebody.

All additional geotechnical data gathered after this from the project should be incorporated into the database and evaluated.

16.1.2 Kamoa Geotechnical Investigation and Design

The geotechnical investigation was based on geotechnical drilling and logging conducted by Ivanhoe mines over the Kamoa project area, which was reviewed and interpreted by SRK and is essentially unchanged from the previous studies (Kamoa 2013 PEA, Kamoa 2016 PFS and Kakula 2016 PEA) in order to provide geotechnical designs for the room-and-pillar method incorporated in the Kamoa 2017 PFS. Geotechnical design for the room-and-pillar method was also carried out by SRK and is also unchanged from the Kamoa 2016 PFS. The controlled convergence room-and-pillar method incorporated in the Kamoa 2017 PFS was developed and designed by Cuprum (2016, 2017a, and 2017b) and reviewed by SRK for the purposes of this report.

16.1.2.1 Geotechnical Database

Geotechnical drilling and logging were conducted by Ivanhoe Mines. SRK completed three site visits to the Kamoa-Kakula Project during 2011 for the purposes of geotechnical and structural logging QA/QC and data quality control.

Findings from the visits have been documented in two memoranda (Jakubec, J. 2010, 2013) which provide outline on-site protocols, quality control reviews, details of the findings, recommendations for future data collection, and update aspects of various geotechnical and mining studies. Limited on-site data analysis and preliminary findings are also documented. Recommendations have been made for regular follow-up visits as the project study level, data quantity, and required level of detail increases.

The geotechnical data collection is acceptable at this stage of the project, but there are a number of areas that require improvement as the project continues:

- Geotechnical data collection: Geotechnical parameter collection is considered to be fair, with ongoing issues noted relating to RQD measurements (inclusion of mechanical breaks). However, the identification of natural versus mechanical breaks is being completed to a high standard. Intact rock strength is locally underestimated; however, in most cases the patterns of strength change are being identified.
- Orientation data collection: Alpha orientation measurements (angle of the break to the core axis) are being collected to a very high standard. Conversely beta measurements (angle of the maximum dip of the fracture related to the reference line) are being collected poorly with errors noted in identification of maximum dip vector, downhole direction, and actual measurement.
- Geotechnical database: The Kamoa geotechnical database was considered to be of fair quality during the audit. While some inherent issues existed, the process of filtering and cleaning the dataset will improve the quality of the geotechnical dataset. SRK understand that significant work has been undertaken recently to improve this.
- Geotechnical recommendations: Several changes have been made to structural and geotechnical data collection processes based on the recommendations by SRK in August 2010 and June 2011. Time should be taken to make sure that these changes are carried out correctly during the early stages of implementation. Additional quality control checks by Kamoa Copper SA geotechnical engineers have been recommended at all stages of data collection.

The status of the structural data being collected has been reviewed. It was decided that the current fault network interpretation cannot be further developed with current information. More detailed structural logging has been recommended and the data capture is underway. Once a more complete set of structural logs are available for the drill core, further interpretation should be undertaken to improve the structural/geotechnical domains.

16.1.2.2 Geotechnical and Structural Models

Structural Domains

The 2012 structural model was updated in 2013/2014 to include the new drilling data and define a primary fault network for the geotechnical studies that could also be used for updating the resource model. The model also needed to be updated in order to reach as near a PFS level of structural understanding as possible, given the scale of the project and lack of outcrop exposure.

During this study, the previously identified faults have been placed into a more robust tectonic framework. The understanding of the age and nature of structural development within the study area has been changed and improved. The new model consists of 45 faults divided into six dominant sets of differing orientations. To assist with the interpretation, other data sources including topographic analysis and surface geophysics were used.

Structural domains together with rock mass data collected are presented in Figure 16.6.

Joint Patterns

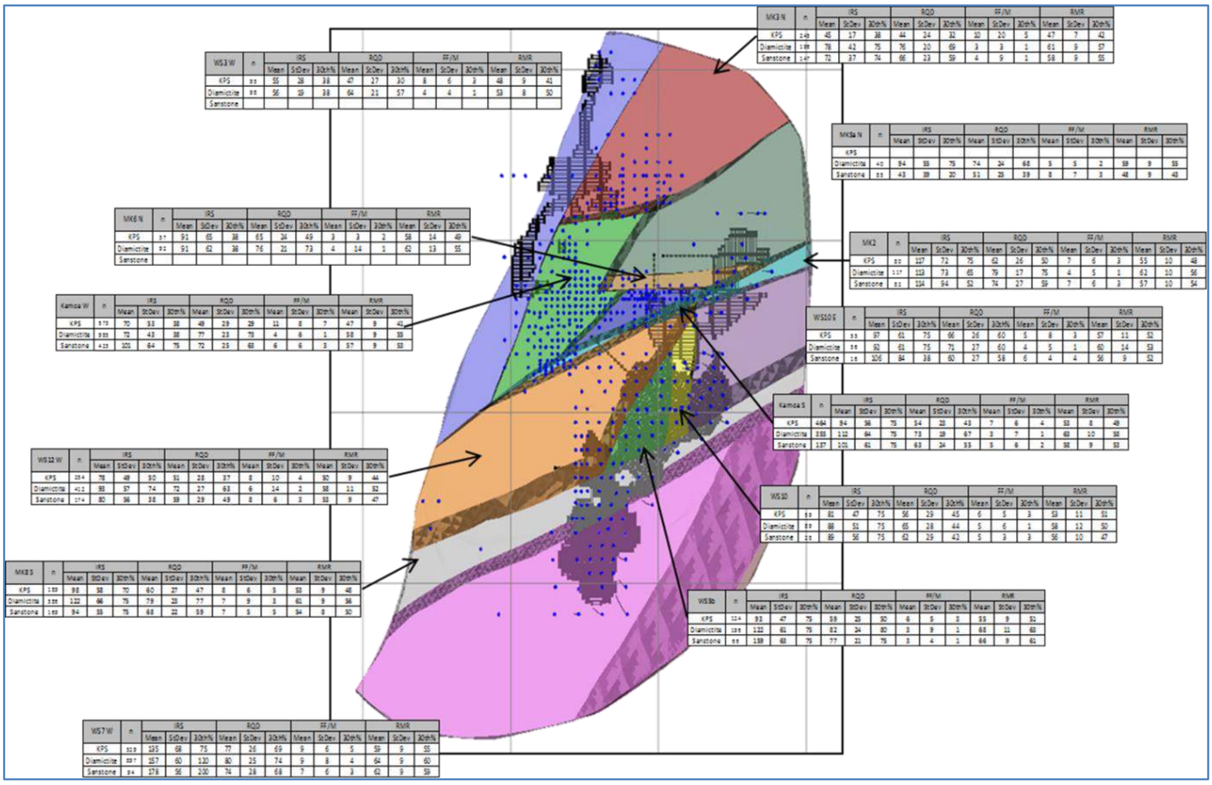
There are 3 main joint sets present across the project area:

1. A steep north-north-east joint set;
2. A shallow dipping set parallel to bedding dip; and
3. A generally steep east-west trend.

The north-north-east striking sub-vertical and shallow bedding plane joint sets are pervasive throughout the area. The east-west joint trend is limited to areas labelled 3A, B, and C in Figure 16.7.

The joint patterns will have a bearing on the anticipated hangingwall deformation and support requirements, pillar strengths and performance characteristics. Therefore, cognisance of the joint patterns is essential during the mining method design. This has been taken into account with the Cuprum mine method design.

Figure 16.6 Structural Domains and Rock Mass Values



MK3 N	n	IRS			RQD			FF/M			RMR		
		Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%
KPS	148	45	17	38	44	24	32	10	20	5	47	7	42
Diamictite	198	78	42	75	76	20	69	3	3	1	61	9	57
Sanstone	147	72	37	74	66	23	59	4	9	1	58	9	55

WS3 W	n	IRS			RQD			FF/M			RMR		
		Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%
KPS	[illegible]	55	28	38	47	27	30	8	6	3	48	9	41
Diamictite	[illegible]	56	19	38	64	21	57	4	4	1	53	8	50
Sanstone													

MK5a N	n	IRS			RQD			FF/M			RMR		
		Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%
KPS													
Diamictite	40	94	55	75	74	24	68	5	5	2	59	9	55
Sanstone	[illegible]	43	39	20	51	25	39	8	7	3	48	9	43

MK6 N	n	IRS			RQD			FF/M			RMR		
		Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%
KPS	37	91	65	38	65	24	49	3	3	2	58	14	49
Diamictite	92	91	62	38	76	21	73	4	14	1	62	13	55
Sanstone													

MK2	n	IRS			RQD			FF/M			RMR		
		Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%
KPS	[illegible]	117	72	75	62	26	50	7	6	3	55	10	48
Diamictite	117	113	73	65	79	17	75	4	5	1	62	10	56
Sanstone	[illegible]	114	94	52	74	27	59	7	6	3	57	10	54

Kamoa W	n	IRS			RQD			FF/M			RMR		
		Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%
KPS	573	70	53	38	49	29	29	11	8	7	47	9	41
Diamictite	[illegible]	72	43	38	77	23	73	4	6	1	58	9	55
Sanstone	423	101	64	75	72	25	63	6	6	3	57	9	53

WS10 E	n	IRS			RQD			FF/M			RMR		
		Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%
KPS	[illegible]	97	61	75	66	26	60	5	8	3	57	11	52
Diamictite	[illegible]	92	61	75	71	27	60	4	5	1	60	14	53
Sanstone	[illegible]	106	84	38	60	27	58	6	4	4	56	9	52

Kamoa S	n	IRS			RQD			FF/M			RMR		
		Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%
KPS	464	94	56	75	54	23	43	7	6	4	53	8	49
Diamictite	355	112	64	75	73	19	67	3	7	1	63	10	58
Sanstone	137	101	61	75	63	24	53	3	6	2	58	9	53

WS12 W	n	IRS			RQD			FF/M			RMR		
		Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%
KPS	234	78	49	50	51	28	37	8	10	4	50	9	44
Diamictite	412	93	57	74	72	27	63	6	14	2	58	11	52
Sanstone	174	80	56	38	59	29	49	8	6	3	53	9	47

WS10	n	IRS			RQD			FF/M			RMR		
		Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%
KPS	[illegible]	81	47	75	56	29	45	6	5	3	53	11	51
Diamictite	[illegible]	88	51	75	65	28	44	5	6	1	58	12	50
Sanstone	[illegible]	89	56	75	62	29	42	5	3	3	56	10	47

MK8 S	n	IRS			RQD			FF/M			RMR		
		Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%
KPS	[illegible]	98	58	70	60	27	47	8	6	5	53	9	48
Diamictite	[illegible]	122	66	75	79	23	77	7	9	3	61	9	56
Sanstone	[illegible]	94	55	75	68	22	59	7	5	5	54	8	50

WS5b	n	IRS			RQD			FF/M			RMR		
		Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%
KPS	114	93	47	75	59	25	50	6	5	3	55	9	51
Diamictite	[illegible]	122	61	75	82	34	80	3	9	1	68	11	63
Sanstone	[illegible]	139	63	75	77	21	75	3	4	1	66	9	61

WS7 W	n	IRS			RQD			FF/M			RMR		
		Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%	Mean	StDev	30th%
KPS	[illegible]	135	68	75	77	26	69	9	6	5	59	9	55
Diamictite	[illegible]	157	60	120	80	25	74	9	8	4	64	9	60
Sanstone	94	178	56	200	74	28	68	7	6	3	62	9	59

Figure 16.7 Overview of Joint Pattern Variations Across the Kamoa Project, Mapped Outline in Background

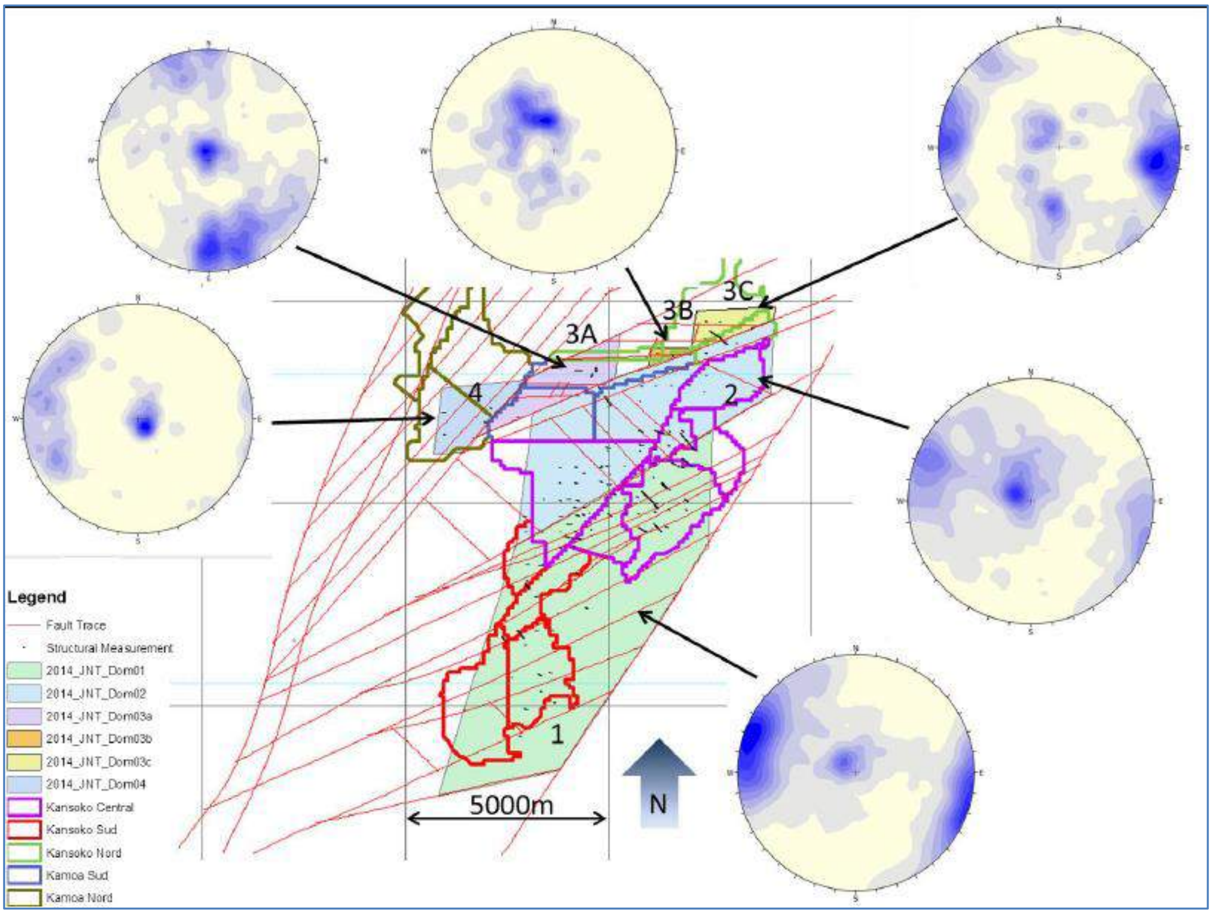


Figure by SRK, 2017.

Weathering

Weathering of the rock mass is highly variable at Kamoa. Structurally controlled weathering appears to extend to considerable depth (over 500 m below surface) in places.

The KPS siltstone is a stratigraphic layer above the orebody, which has a tendency to weather rapidly and the distance between the roof and this layer varies considerably and in some areas, it forms the hangingwall to the deposit. A safe distance between the roof of the mining operation and the KPS should be maintained to a minimum of 3 m.

16.1.2.3 Geotechnical Domains

The assessment of the geotechnical properties assumes three primary lithological domains, namely siltstone, diamictite and sandstone (from the Roan basement rock). The orebody is located primarily within the diamictite. An additional geotechnical domain can be defined that consists of the weathered rock mass at surface. SRK modelled the base of the weathered rock based on the weathering descriptions in the provided drill logs.

A statistical approach was used to evaluate the data (separated by lithology), resulting in primary geotechnical division of the rock mass based on weathering. The weathering category was used to establish fresh, moderately weathered, and extremely weathered geotechnical domains. 3D wireframes were developed with average thickness of 10 m (extremely weathered) and 45 m (moderately weathered).

The established structural domains were used to further subdivide the data, with four fresh rock geotechnical domains established. The near surface Extremely Weathered Geotechnical Domain was not considered further for the underground geotechnical study.

The geotechnical parameters for intact rock strength, RQD, fracture frequency, joint condition rating, and RMR 89 for each geotechnical domain are presented in Table 16.3. Figure 16.8 shows a plan view of three fresh geotechnical domains: north, central, and south.

Figure 16.8 Plan View of Three Fresh Geotechnical Domains (North, Central, South)

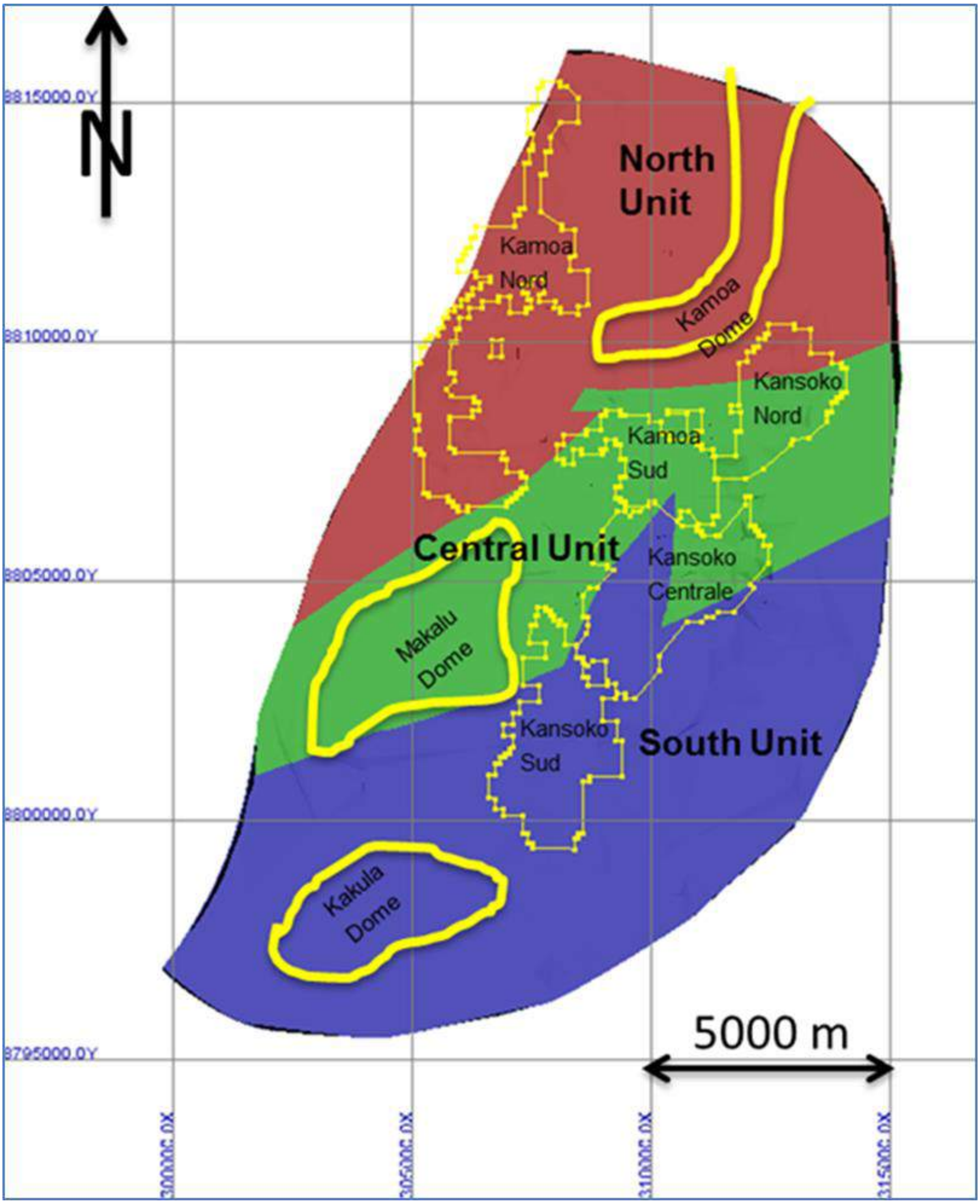


Figure by SRK, 2017.

Table 16.3 Summary of Geotechnical Parameters per Geotechnical Domain

Domain	Stratigraphy	RQD (%)	FF/m	RMR_{89}	Intact Young's Modulus (GPa)	Poisson's Ratio
Moderately Weathered	KPS	40	10	43	35 Est	0.24 Est
	Diamictite	63	6	51	47	0.28
	Sandstone	55	8	48	32	0.22
Fresh, North	KPS	48	11	47	66	0.28
	Diamictite	76	4	58	67	0.27
	Sandstone	67	6	56	58	0.23
Fresh, Central	KPS	55	7	53	66	0.28
	Diamictite	73	5	60	67	0.27
	Sandstone	62	6	55	58	0.23
Fresh, South	KPS	68	8	56	66	0.28
	Diamictite	80	8	63	67	0.27
	Sandstone	71	6	59	58	0.23

Rock Properties

Geomechanical laboratory testing was undertaken during 2012 and 2013 by SRK Canada a total of 121 samples were tested to determine UCS. The engineered intact rock strength (IRS) presented in Table 16.4 considers the field estimated IRS (as logged by African Mining Consultants), field point load testing, and laboratory unconfined compressive strength testing. Table 16.4 lists the mean values, standard deviation (in brackets), and the derived engineered intact UCS of each stratigraphy within geotechnical domains. Due to a lack of data coverage across the deposit, the UCS data has been repeated in each domain for comparison to other data sources.

Table 16.4 Summary of Intact Rock Strength Estimates per Geotechnical Domain (Standard Deviation in Parenthesis)

Domain	Stratigraphy	Logged IRS (MPa)	# of UCS Tests	Intact UCS (MPa)	Point Load Test (MPa)		Engineered IRS (MPa)
					Axial	Diametral	
Weathered	KPS	45 (32)	1	123	75 (51)	61 (42)	45
	Diamictite	44 (30)	8	56 (31)	66 (54)	50 (30)	50
	Sandstone	44 (31)	4	153 (48)	88 (56)	81 (52)	75
Fresh, North	KPS	63 (47)	8	208 (36)	67 (43)	66 (44)	90
	Diamictite	72 (42)	17	98 (29)	97 (55)	71 (35)	100
	Sandstone	86 (59)	2	219 (22)	96 (55)	86 (59)	100
Fresh, Central	KPS	91 (57)	8	208 (36)	115 (54)	108 (54)	90
	Diamictite	101 (62)	17	98 (29)	80 (44)	92 (43)	100
	Sandstone	91 (63)	2	219 (22)	132 (50)	112 (60)	125
Fresh, South	KPS	116 (49)	8	208 (36)	144 (66)	140 (56)	120
	Diamictite	143 (64)	17	98 (29)	108 (48)	106 (39)	125
	Sandstone	131 (69)	2	219 (22)	121 (60)	126 (49)	125

Furthermore, a thorough laboratory testing programme was undertaken by Cuprum in 2015 to establish an extended stress – strain correlation in the post failure phase of the diamictite (mineralised zone). The laboratory testing established the average post failure strength for siltstone was approximately 14% of the UCS for siltstone and 16.5% for diamictite. It is noted that these stress – strain correlations correspond well with the dolomite rock mass being extracted according to Cuprum (2017) at a number of copper mines in Poland are using the controlled convergence room-and-pillar mining method.

16.1.2.4 Geotechnical Design

The two mining methods to be used are room-and-pillar and controlled convergence room-and-pillar.

The room-and-pillar design provides for a stiff, non-yielding system in which excavations remain open for the LOM and primary infrastructure and access ways (declines and strike drives) are accessible without interruptions, all the way from the mining front back towards the centre of the mine. However, this mining method significantly reduces the extraction ratios.

Subsequently, a strategic decision to change the mining method was made by Ivanhoe, after a visit to KGHM Polska Miedź S.A in Poland to view a controlled convergence room-and-pillar mining method used by their Polish copper mines. This mining method is used by a number of copper mines in Poland, which therefore provide a basis for comparison with regard to attributes and efficacy of the method. This approach provides for a "controlled" goafing of the back area under the action of smooth (continuous) hangingwall closure, and rests on the principle of crushed pillars providing a residual support capacity directly after being cut at the advancing face.

16.1.2.5 Geotechnical Discussion of Mining Methods

Room-and-pillar will be used up to a depth of 150 m, to limit the risk of subsidence. There is abundant experience in the application of room-and-pillar mining to tabular orebodies in a wide range of geological environments. It is notable that large-scale room-and-pillar mining have been associated with unexpected massive collapses due to sudden failures over an extensive area. The key requirements for successful application of the room-and-pillar method is a proper understanding of the stability of the rooms and ensuring that the in-panel pillar layouts are adequate for the expected conditions. Taking this into account the room-and-pillar mining method has been adequately designed.

The controlled convergence room-and-pillar mining method has an aggressive in-panel recovery layout where pillars are designed on experience yield progressively as the mining advances. The planned application of the controlled convergence room-and-pillar method rests on the premise that the same method has been successful at mines in Poland owned by KGHM Polska Miedz SA (KGHM) (Cuprum, 2017a). This premise has been thoroughly assessed by Cuprum (2015) where the following geotechnical parameters for the hangingwall, orebody and footwall have been compared:

- Geology (orogeny, stratigraphy).
- Rock mass strength and performance characteristics (laboratory tests, rock types, rock mass classification, local tectonic disturbances).

It should be noted that the risk to the roof stability that exists in the KGHM mines also exists in the Kansoko mining area. The problems encountered and strategies applied to mitigate against these problems at KGHM's Polish mines have been recommended for the Kansoko Mine.

The layout is geometrically well-defined with dip and strike barrier pillars and panels up to 300 m wide and 500 m long.

Overall, the controlled convergence room-and-pillar methodology appears to be suitable for the Kansoko mining area, however, a focussed assessment of the hangingwall conditions must be carried out in the next phase of study. In addition, the underground development and trial stoping is required to better understand the geological structure and its potential influence on the mine design.

Pillar Design (Room-and-Pillar)

Pillar strength for the room-and-pillar mining method has been designed adequately to provide for a stiff, non-yielding system for the areas where this mining method is to be used.

The pillar design and theoretical extraction ratios for a range of depth intervals and mining heights were based on Tributary Area Theory (TAT) for square and rectangular pillars. The Kamoa resource and surrounding rock conditions change significantly across the project area. A variety of pillar designs are provided to accommodate the changing rock mass conditions.

Geotechnical logging and laboratory tests were used to derive a Design Rock Mass Strength (DRMS) equivalent to the strength of 1 m^3 in the pillar. Laubscher's 1990 method was used to determine the DRMS.

In-panel pillar designs are based on the Hedley and Grant (1972) empirical formula. The formula derives the in-situ pillar strength from the DRMS and the pillar dimensions. The strength of the stability pillars was assessed based on the empirical relationship after Stacey and Page, considering a panel length (500 m) and a width of 40 m.

The in-panel pillar loads were initially calculated using the Tributary Area Theory (TAT) that assumes that pillars carry the entire load to surface and this is shared equally by all the pillars. The pillar load is a function of the virgin vertical stress and extraction ratio.

The extraction ratios calculated by Cuprum for the room-and-pillar mining area are shown in Table 16.5.

Table 16.5 Room-and-Pillar Mining Method Extraction Ratios

Dip Intervals (°)	Mining Height Intervals in Panel (m)	Extraction Ratio (60–150 m) (%)	Extraction Ratio (150–250 m) (%)
0–12	3 to ≤4	79	75
13–16		73	71
17–20		74	71
21–25		72	72
26–30		71	69
31–35		69	66
0–12	<4 to ≤5	75	72
13–16		72	69
17–20		72	70
21–25		66	64
26–30		64	62
31–35		63	60
0–12	<5 to ≤6	72	69
13–16		70	67
17–20		70	68
21–25		64	62
26–30		63	61
31–35		62	59

Pillar Design (Controlled Convergence Room-and-Pillar)

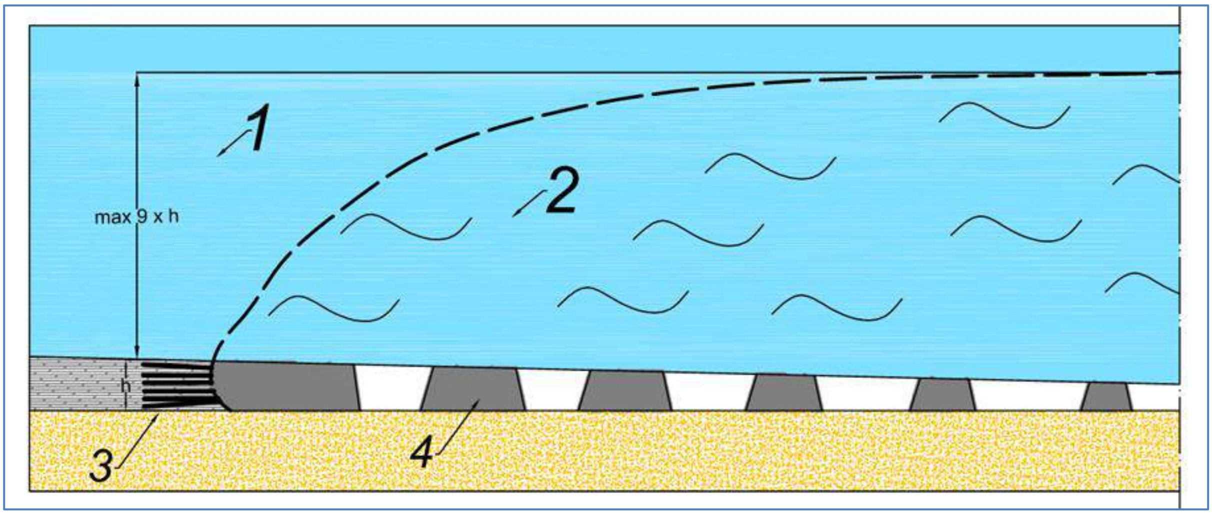
Pillar strength for the controlled convergence room-and-pillar mining method is premised on a percentage of the UCS for the post-failure strength estimate. The results of 14% and 16.5% of the UCS are reasonable quantities (~20 MPa) post failure strength. This is the strength assigned to the pillars (or rather, the "pillar cores") for the pillar design. The extraction ratio for the controlled convergence room-and-pillar mining method are shown in Table 16.6.

The anticipated mode of failure/deformation of the hangingwall appears to be a controlled closure of a continuous stratigraphic horizon as the back-area pillars deform in post failure mode as depicted in Figure 16.9. The Cuprum (2016) report does not specifically state that it takes into account the potential influence of structural discontinuities that may result in wedge or structural failure or how the outreach of the distressed rock mass area will enable smooth roof bending strata. However, these scenarios are recognised and provision made in the form of recommendations that hydraulic props, wooden cribs and cable bolts must be used in areas where complex geological or difficult mining conditions exist.

Table 16.6 Controlled Convergence Room-and-Pillar Mining Method Extraction Ratios

Dip Intervals (°)	Mining Height Intervals in Panel (m)	Extraction Ratio (Primary Phase = Face Blasting Works) (%)	Extraction Ratio Secondary (From Pillars Scraping) (%)	Total In-Panel Extraction (%)
0–12	3 to ≤4	66	24	90
13–16		65	24	89
17–20		64	25	89
21–25		62	27	89
26–30		52	33	85
31–35		49	36	85
0–12	<4 to ≤5	62	27	89
13–16		60	28	88
17–20		59	29	88
21–25		56	31	87
26–30		46	38	84
31–35		44	40	84
0–12	<5 to ≤6	56	31	87
13–16		55	31	86
17–20		55	32	87
21–25		53	33	86
26–30		44	39	83
31–35		42	41	83

Figure 16.9 Controlled Convergence Room-and-Pillar Rock Mass Impact



1 – rock mass prior to extraction; 2 – destressed and delaminated rock mass; 3 - blasting holes; 4 - primary pillars. Figure by KGHM Cuprum, 2017.

Protection of Main Access Ways

Cuprum (2016) has stipulated that the declines and underground chambers will be protected using 20 m protection pillars on either side of the decline array and underground chambers where the workings are shallower than 600 m. The width of this pillar will increase to 40 m for depths below 600 m. Figure 16.10 and Figure 16.11 show the protection pillars for declines of mining dip <12° and 13° to 16°, respectively.

The scenarios in both Figure 16.10 and Figure 16.11 show that mining progresses towards the declines and this is in effect, a retreat mining methodology. The secondary development is only required ahead of the mining faces and should not be required in the back areas where the controlled convergence room-and-pillar has occurred. There is however a concern when reviewing the mine design/mining direction that the mining direction is away from the main access decline. In this situation additional pillars may be required on the goafing side of the secondary access ways to protect these access ways for the life of the panel specifically to contain any unravelling, wedge failure which may result in closure of these access ways. This has been taken into account in the Cuprum (2017b) report where it is stated that a protection zone will be required for the secondary drives that are required for access to other mining panels. Provided that the protection zone (including that applicable to secondary drives) is implemented, then it can be concluded that the access ways are adequately protected.

Figure 16.10 Controlled Convergence Room-and-Pillar Mining Method and Pillar Geometry for a Deposit Dip up to 12°

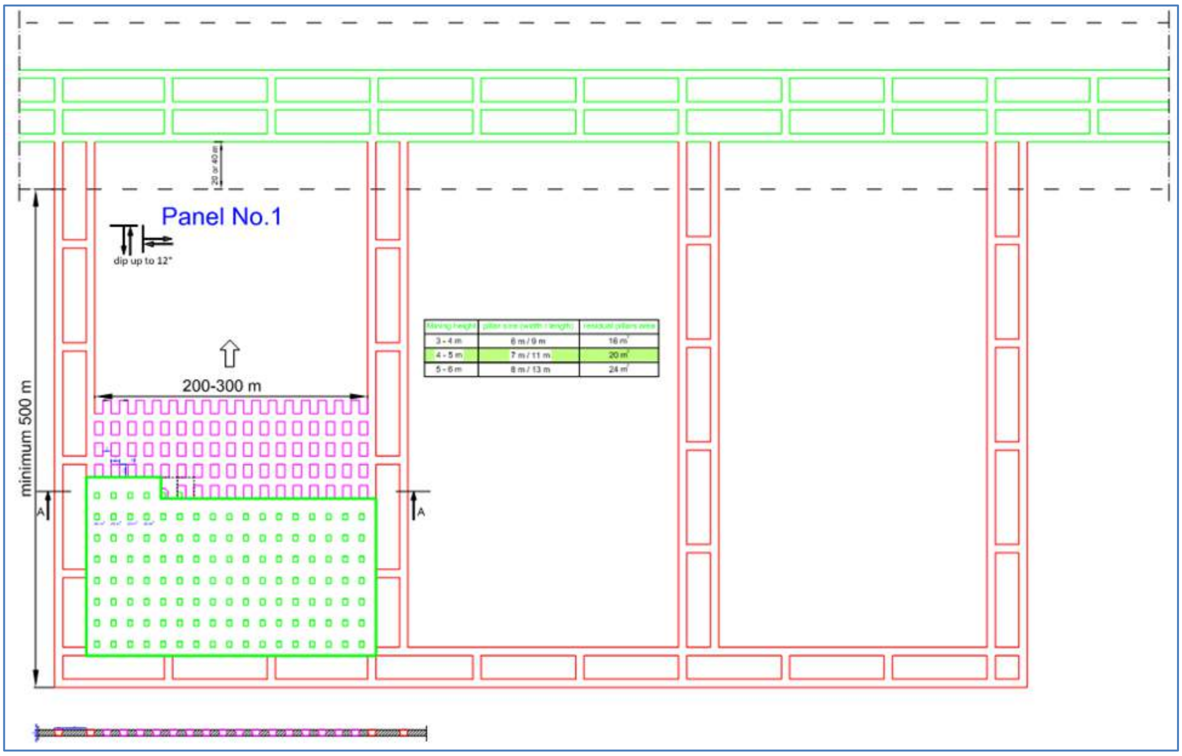


Figure by KGHM Cuprum, 2017.

Figure 16.11 Controlled Convergence Room-and-Pillar Mining Method and Pillar Geometry for a Deposit with Dip Angle of 13 to 16°

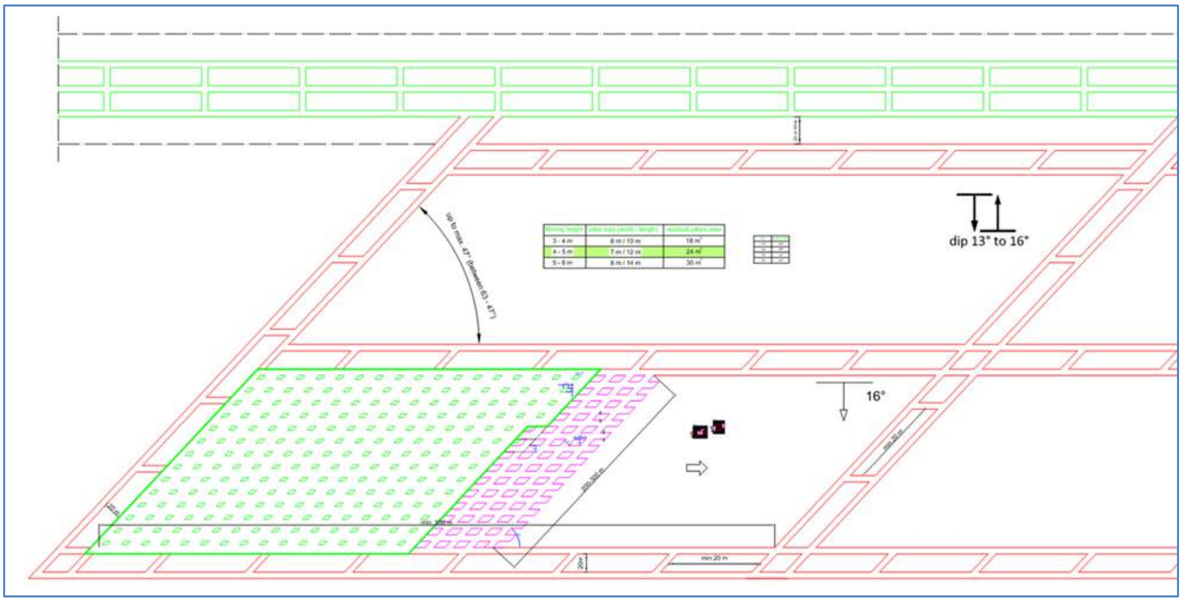


Figure by KGHM Cuprum, 2017.

Numerical Analysis

The two-dimensional software program Phase 2 (version 8) was used by Cuprum to conduct numerical modelling to determine the minimum width of the protection/safety pillars. An elastic, perfectly plastic material for the roof, orebody and the footwall were used. The mining in the vicinity of a long-term excavation (main access way, etc.) were included in an attempt to establish both the size of the required protection/barrier pillar and the overall stability of the excavation. The input parameters were appropriately determined using the laboratory test results and Rocklab software.

The modelling exercise and software code is suitable to determine the safety pillar dimensions and the height of the tensile zone of the long-term excavation at this level of study.

It must be borne in mind that that all models are a simplification of reality and consequently, the residual load applied to the barrier pillars is not completely understood. It is important to estimate or anticipate the potential for pillar punching into the footwall, seismicity (bursting of the pillars), shear failure of the hangingwall around the pillars and minimum pillar sizes required for the barrier (protection) pillars. Similarly, the height effect of irregular geometries (i.e. where one side is significantly taller than another) needs to be investigated in the next phase of study.

The actual response of the hangingwall, footwall and in-panel pillars needs to be quantified during a mining trial. Detailed monitoring as described by Cuprum (2016) will be essential to obtain the maximum benefit of the trial. This data then needs to be feed into a 3D in-elastic numerical model such as FLAC to obtain a better understanding of the rock mass behaviour associated with this mining method.

Support Design

Support design uses the principle of the height of a de-stressed “tensile” zone within the hangingwall above the exposed room and a suitable factor of safety (FoS). Cuprum (2016) typically recommend tendons in the order of 2.0 m and spaced apart at acceptable intervals. At this stage of the study, the review finds that the support recommendations appear to be reasonable but can be adjusted as necessary going forward.

However, where the back-area pillars will continue to lose strength and hangingwall closure will increase, it is important to understand if the hangingwall will behave as a continuous stratigraphic horizon. If so, tendons of length, spacing and capacity are adequately designed as a function of the tensile zone above the excavated rooms.

Alternatively, if the hangingwall does not behave as a continuous stratigraphic horizon but is expected to unravel or undergo wedge failure or parting on a stratigraphic contact, wide-scale falls of ground (FoGs) may occur in the back area. Such FoGs may either reduce the extraction ratio and/or result in catastrophic closure of critical accesses if the failure is not managed proactively. It is understood that the project will be carried out under a comprehensive monitoring programme, as a trial, and the mining direction, additional support and monitoring systems appear to have been adapted to anticipate and manage this mode of failure.

Preliminary Subsidence Review

Surface deformations will begin to reveal themselves in the first five years after mining begins in the area located to the south of the mine buildings. At first, two local troughs will develop, and the maximum surface subsidence will not exceed 0.4 m. The impact of drainage on surface deformation will be negligible during the first five years. By analysing the picture of total surface subsidence in the target period, it can be stated that a large field panel of displacement with two local centres will be created at the mine site of the designed mine, which will form themselves over both mine regions (Centrale and Sud). The larger trough will develop in the area of the Centrale region and its maximum subsidence will be about 2.7 m. What is important, this trough will be steep but strongly restricted spatially from the west side of the main plant buildings.

The displacement connected with of the south area (Sud region) should not exceed a maximum of 2.6 m. From the north, this local displacement field panel will exhibit significant slope of the trough profile coming up to 10.0 mm/m, accompanied by horizontal deformations with the maximum values of ±5.5 mm/m.

Water Management

This is discussed in the Cuprum (2016) report however the management of water is not specifically addressed. The goafing of the hangingwall will significantly increase permeability of the overlying rock mass. Golder's groundwater model takes cognisance of a goafing zone, equal to 2 x mining height and a damage zone, equal to 9 x mining height. They report an increase of inflow due to goafing of 12% to 30% in different modelled scenarios. It is important that this is taken into consideration in the design.

16.1.2.6 Conclusions and Potential Geotechnical Risks

The geotechnical data has been collected according to internationally acceptable standards and QA/QC reviews were done onsite to confirm compliance to data collection standards. Rock material testing of the main lithological has been done to establish typical rock mass strengths and elastic properties. Overall the work done is suitable for PFS requirements.

The geotechnical risks for this project were identified and are summarised below:

- The uncertainty due to the wide-spacing of data and lack of understanding of the frequency of structures and their deformation zones, that may impact the competency of the underground rock mass and the continuity of the deposit.
- The actual response of the hangingwall, footwall and in-panel pillars needs to be quantified during a mining trial. Detailed monitoring as described by Cuprum (2017a) will be essential to obtain the maximum benefit of the trial.
- The stress environment is unknown at this stage and it appears that Cuprum has used a k ratio of 1. However, if the horizontal stress is significantly high this will could result in an increase in the depth of failure in the hangingwall of the long-term excavations.
- The pillar design and extraction percentages are based on the summarised data obtained from drillhole core only. The information is considered representative but needs to be verified through data collection from underground exposures.
- Good quality conventional blasting appears to have been assumed in Cuprum's analyses (2016, 2017a, and 2017b), where limited overbreak occurs. Poor blasting results in smaller and taller- than designed pillars, negatively impacting on the pillar and span stability. Failure to achieve good quality blasting will significantly affect pillar performance.
- In portions of the various mining areas, the KPS lithology has been interpreted to possibly form the hangingwall package, and in some situations the upper portions of the pillars. Fresh KPS is not considered to be a concern for the rock mass, but exposed and weathered pyritic siltstone could rapidly degrade and cause significant hangingwall stability problems.
- The stability of the rock mass within the mining environment is not well known, specifically with respect to geological structures contained in the pillars and the hangingwall.
- It is important to retreat towards a stable access during pillar extraction to ensure safety of personnel.

16.1.2.7 Recommendations for Feasibility Study

The geology in the area includes significant geological structure with numerous faults and a wide range of joint orientations. The occurrence and the condition of these structures needs to be better understood. The behaviour of the hangingwall will be affected by geological structure and orientation. It is recommended that a full scale geotechnical mapping of the rock mass is done during the development and trial mining phase.

The main requirements for the successful implementation of the controlled convergence room-and-pillar mining method includes a good understanding of the hangingwall stability of the rooms during primary and secondary stages as well as confidence that the post peak performance of crush pillars is adequate for the expected conditions. The in-panel recovery layout is aggressive and although it appears to be suitable for the Kansoko deposit concerns have been raised by Cuprum (Cuprum 2017a, page 92) about the potential for a cave in hazard. Cuprum however, has carried out a thorough comparison of the geological and geotechnical parameters for the KHGM mines and concluded that the Kansoko rock mass was similar to that in Poland. SRK's review of this work indicates that this is a reasonable assumption and suitable for a PFS.

The stability/barrier pillars must be large enough to prevent undue damage to the long-term access ways and major infrastructure. The numerical modelling done is suitable for this level of study and indicates that the required size of the stability pillars and fulfils the requirements of a PFS.

A proper understanding of the pillar strength and post failure strength of the pillar is critical to the project to ensure both stability and facilitate the required extraction.

The actual response of the hangingwall, footwall and in-panel pillars needs to be quantified during a mining trial. Detailed structural and geotechnical mapping and geotechnical instrumentation programme in the proposed trial site as described by Cuprum will be essential to obtain the maximum benefit of the trial.

The expected deformability of the pillars, the stability of the barrier pillars, strengths and expectation of the main access ways to remain open for the life of the panel needs to be quantified during the next stage of the study using data obtained during the trial mining process.

The stress environment is unknown at this stage and it appears that Cuprum used a k ratio of 1. However, if the horizontal stress is significantly high this could result in an increase in the depth of failure in the hangingwall of the long-term excavations. It is recommended that initially the modelling is done with varying k ratios to determine the potential effect of high horizontal stress. In addition to this it is recommended that during the trial period stress measurement is done to establish the magnitude and direction of the virgin stress.

The layout and sequences may need to be optimised in the next phase of study to ensure safety of personnel.

16.2 Underground Mining

16.2.1 Kakula Underground Mining

16.2.1.1 Introduction

Based on updated design criteria, Stantec optimised the mining methods, mine design, and schedule recommendations from previous studies. Mining methods selection focused on high productivity methods with an emphasis on maximizing ore recoveries and production grades, while reducing operating costs. The mine schedule focused on optimizing mining block sequencing, maximizing grades in the early years, and removing development from the critical path. The following subsections discuss the mining methods selection process and the resultant mine designs and schedules.

The mining methods for the Kakula deposit are drift-and-fill and room-and-pillar. Drift-and-fill represents the majority of the mining for the Kakula deposit. The room-and-pillar area represents just over 1% of the Probable Mineral Reserve and will mainly be used for early ore production while the drift-and-fill areas are being developed. The Kakula Mineral Reserve by mining method is summarised in Table 16.7.

Table 16.7 Kakula Probable Mineral Reserves by Mining Method

Production by Mining Method	**Ore (Mt)**	**NSR20 (\$/t)**	**TCu (%)**	**Fe (%)**	**As (%)**	**S (%)**
Ore Development	3.9	201.43	4.66	4.88	0.00	1.28
Drift-and-Fill	113.9	241.00	5.54	4.83	0.00	1.53
Room-and-Pillar	2.0	163.41	3.81	4.76	–	0.92
Total Ore*	**119.7**	**238.44**	**5.48**	**4.83**	**0.00**	**1.51**

*May not sum to total due to rounding.

16.2.1.2 Mine Design Parameters

Ore and Waste Properties

The Kakula deposit is a large stratiform copper deposit, typical of sediment-hosted deposits. It is tabular, with dips varying from 0° to 58° and thicknesses varying from 3 m to 18 m (averaging 8.66 m at a \$100 NSR cut-off). The ore zone density has been defined as using a plus 3% TCu cut-off. The swell factor for development is 50%.

Table 16.8 details the geological engineering parameters of the ore and surrounding waste rock of the Kakula deposit.

Table 16.8 Bulk Density/In Situ by Area

Bulk Density/In-Situ	Min (t/m³)	Max (t/m³)	Average (t/m³)
Ore	2.27	3.23	2.81
Hangingwall	2.39	3.18	2.80
Footwall	2.21	3.04	2.67

Mine Planning

Development

All lateral development (e.g., infrastructure access), unless otherwise specified, will be 5.5 m W x 6 m H, with 1.5 m arch corners.

Conveyor drifts will have a maximum gradient of ±10°. They will be 7 m W x 6 m H (1.5 m arch corners), with remucks located every 150 m.

All perimeter drifts will be 5.5 m W x 6 m H (1.5 m arch corners) and have a maximum gradient of ±8.5°. Perimeter service drift development will consist of 2 parallel drifts with cross-cuts every 150 m. The cross-cuts will have a flat back.

All connection drifts connecting the north and south perimeter service drifts will be 6 m W x 6 m H (1.5 m arch corners, with a flat back) and will have a maximum gradient of ±8.5°.

Vertical development consists of ventilation raises, bins, and boreholes. All ventilation raises are planned to be excavated with a raisebore drill. If there is an opportunity, blind sinking will be considered. VS1 will be 5.5 m in diameter, VS2 will be 6 m in diameter, and VS3 will be 4 m in diameter. All other raises will be 6 m in diameter. Bins will be excavated as drop raises using longhole drills. Boreholes for paste fill or services to the underground will be drilled from surface using surface drills. These boreholes will be cased as required for their purpose.

Room-and-Pillar

Room-and-pillar mining will be the early mining area for the Kakula deposit. The mining panel width and length limits will be determined by production requirements. The height of the panel will be from 3 m (measured on the central axis with a minimum wall height of 2.5 m) to 6 m. The height will vary depending on the dip of ore deposit and mining thickness. Main drifts will run parallel to the strike of the panel for dips less than 20°, with cross-cut drifts running at an acute angle to the mains to ensure the grade of the main drifts remain less than 12°. For dips less than 12°, cross-cuts will be oriented perpendicular to the mains. For dips greater than 12° (up to 35°), pillars will be diamond-shaped. Cross-section shapes will be rectangular (vertical on sidewalls) with a shanty hangingwall. The angle of the shanty will depend upon the strata dip. The width will be 6 m in plan view. Pillar sizes are identified in Table 16.9.

Table 16.9 Pillar Criteria for 3 m to 4 m Orebody Thickness for 30 MPa Design Rock Mass Strength at 300 m Depth

Orebody dip (°)	Dip pillar width (m)	Strike pillar length (m)	Dip angle (°)	Acute angle (°)	Extraction ratio (%)	W_e:H ratio	Pillar stress (MPa)	Slender pillar strength (MPa)	Squat pillar strength (MPa)	FoS
0–12	9	10	12	90	62	2.4	21	33	44	1.53
>13–16	10	13	16	57	62	2.4	21	33	43	1.53
>17–20	14	16	20	40	62	2.4	21	33	42	1.55
>21–25	10	11	25	54	60	2.2	20	31	43	1.56
>26–30	10	12	30	43	58	1.9	19	29	44	1.51
>31–35	12	12	35	35	56	1.8	18	28	44	1.55

Drift-and-Fill

The majority of mining will be drift-and-fill. A typical mining block with production drifts perpendicular to the connection drift will be 216 m wide. There will be no barrier pillars required between blocks. The orebody width will determine the maximum length of the mining block. Production drift cross-section shapes will consist of vertical walls, and the hangingwall width will be a maximum of 6 m. The minimum mining height will be 2.5 m (3 m measured on the central axis). The maximum mining height will be 6 m measured on the central axis. Each mining block will consist of 3 mining units. Each mining unit will comprise 4 primary, 4 secondary, and 4 tertiary headings. For dips less than 12°, drifts and cross-cuts will be oriented perpendicularly. For steep-dipping deposits (greater than 12°), cross-cuts will be angled such that development inclination grade does not exceed its maximum limit (12°).

Production Pillars

Drift-and-fill production pillar sizes are presented in Table 16.2 and pillar orientation in Figure 16.12.

Figure 16.12 Drift-and-Fill Design (Plan View)

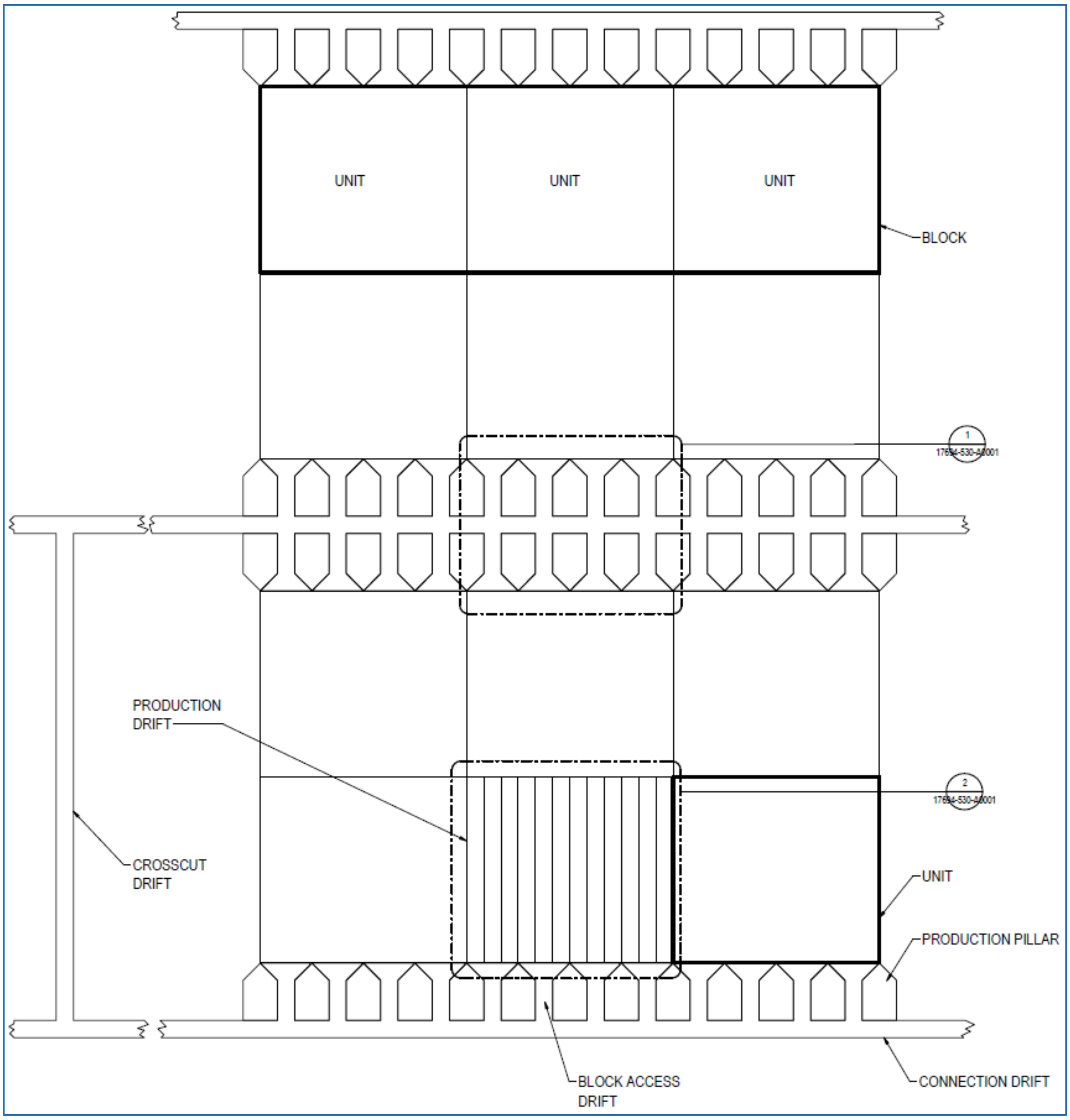


Figure by Stantec, 2019.

Production Development

A comprehensive design criteria list was prepared by Stantec. Geotechnical information was used for development designs, ground support, and mining block layouts. The ventilation parameters defined minimum air velocities, air quantities, and cooling requirements. The underground infrastructure parameters defined required mine facilities, excavation sizes, and equipment requirements (mobile and fixed).

Mining Method Selection

The mining methods for the Kakula deposit have been modified from previous studies. Initial mining will be completed using the room-and-pillar mining method. The majority of the deposit (greater than 98% of the targeted mineral resource) will be mined by drift-and-fill.

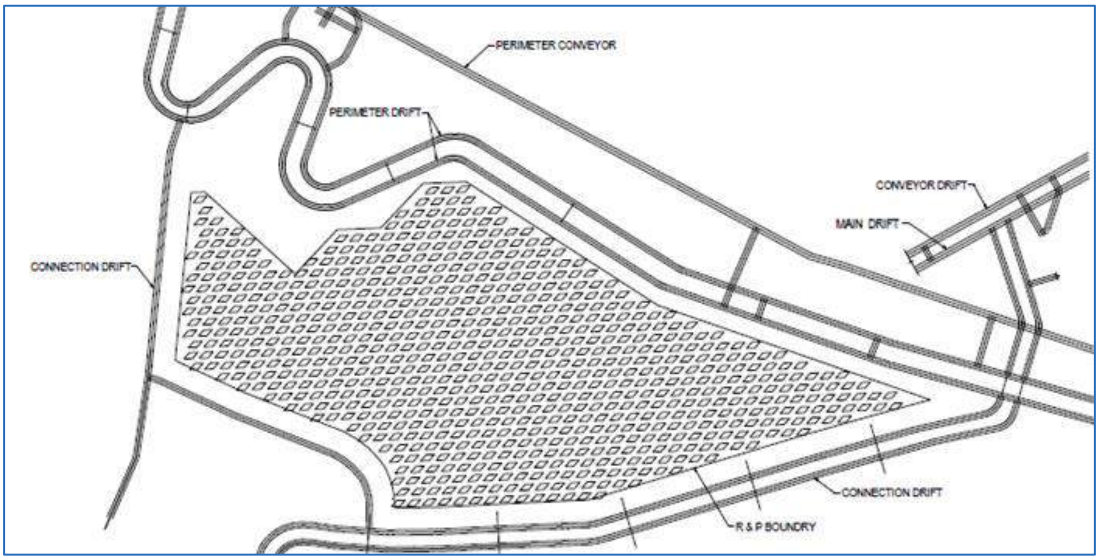
Controlled convergence room-and-pillar was excluded from this Study as a mining method option due to several factors such as the orebody thickness and lower extraction ratios.

Mining Shapes Design

Room-and-Pillar

Room-and-pillar has been designed for the deposit as shown in Figure 16.13. Ore thickness ranges from 3 m to 6 m and will provide early ore production while the drift-and-fill areas are being developed. Production development will be in a grid-like fashion through a series of main and cross-cut drifts. The room-and-pillar area represents just over 1% of the probable mineral reserve tonnage (1,976 kt).

Figure 16.13 Early Ore Room-and-Pillar Location



Main drifts will run parallel to the strike of the panel for dips less than 20°, with cross-cut drifts running at an acute angle to the mains to ensure the grade of the cross-cuts remain less than 12°. Where the dip is greater than 20°, the main drifts will be developed slightly off the strike to accommodate the acute angle between the main and the cross-cuts. This will ensure the pillars have the required area to maintain long-term stability in the room-and-pillar panels.

The room-and-pillar area has been designed to prevent subsidence and will be accessible over the mine life if flow-through ventilation is maintained.

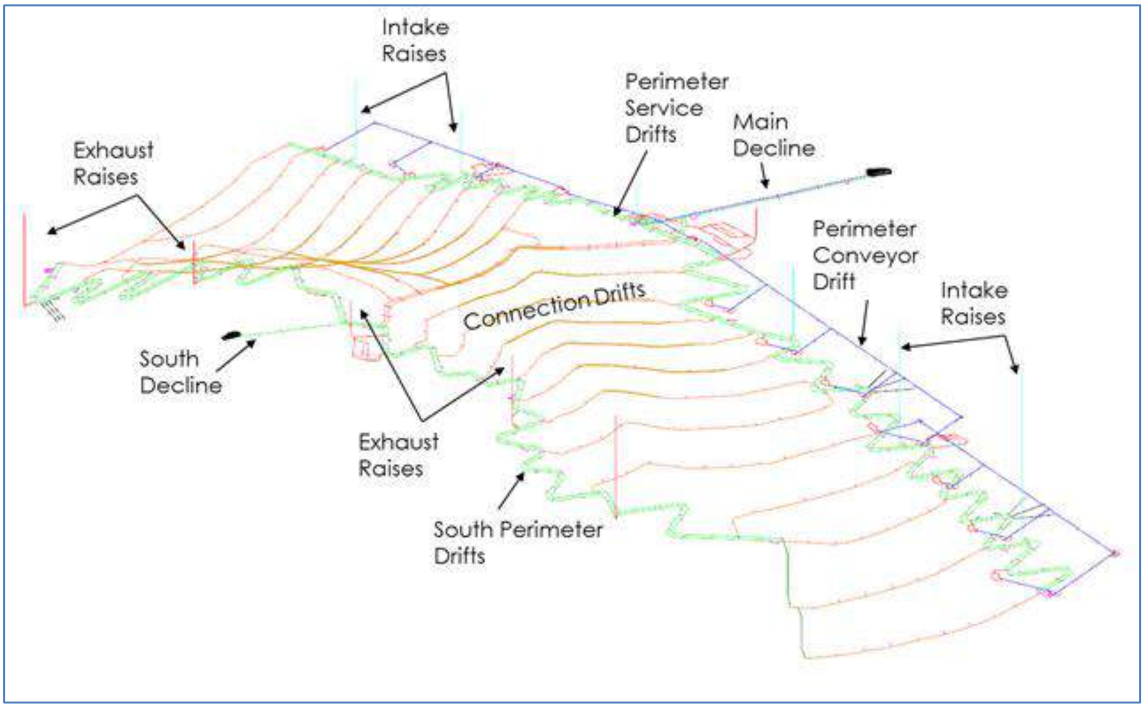
Drift-and-Fill

Drift-and-fill mining will be the primary method of extraction for the Kakula deposit. To establish the mining method, a pair of twin perimeter drifts will first be driven around the north and south margins of the defined ore zone. Connection drifts will then be driven on strike between the perimeter drifts (see Figure 16.14). The distance between the connection drift centers is dependent on the dip and the depth of the zones. The connection drifts will be driven on the footwall of the ore horizon and will provide the framework for defining the drift-and-fill mining blocks and accesses for equipment and ventilation to the block access and production drifts.

Where the dip of the orebody is less than 20°, connection drifts will be driven on strike, with the block access and production headings driven up and down dip from the connection drifts. Where dips are greater than 12°, the block access and production headings will be driven at acute angles to the connection drifts to maintain a maximum gradient of 12°. Where the dip of the orebody is greater than 20°, the connection drift will be driven slightly off strike to maintain a maximum grade of 8.5° and to minimise the acute angle between connection drifts and the block access and production headings.

Connection drifts will divide the orebody into mining blocks. There will be a mining block on either side of the connection drift, and each mining block will contain three mining units. Drift-and-fill mining and the associated connection drifts account for 117,752 kt of the probable mineral reserve (113,878 kt drift-and-fill and 3,874 kt from connection drifts).

Figure 16.14 Mine Development Framework



A mining unit will consist of 4 primary, 4 secondary, and 4 tertiary production headings. Each series of the 3 heading types will be driven from a block access drift. The resulting production pillars will stay in place until full extraction of the mining block has been completed and filled.

The production headings will be 6 m wide, divided into primary, secondary, and tertiary headings, and will only contain temporary services required for advancement and backfilling. Their relative angle to the connection drift will change based on the dip of the orebody. The maximum gradient for these drifts is 12°, and they will be driven to a point half the distance between the connection drifts.

Primary drifts will be driven to the designed length, paste-filled, and cured prior to secondary drift development. This process will continue through tertiary extraction.

The remaining production pillars will be extracted sequentially on a retreat toward the north or south perimeter drifts. Once the initial pillars have been taken and filled, heading ventilation for subsequent pillar extraction will be ducted to the face from the perimeter drifts. Figure 16.15 illustrates the extraction sequence.

Figure 16.15 Drift-and-Fill Extraction Sequence

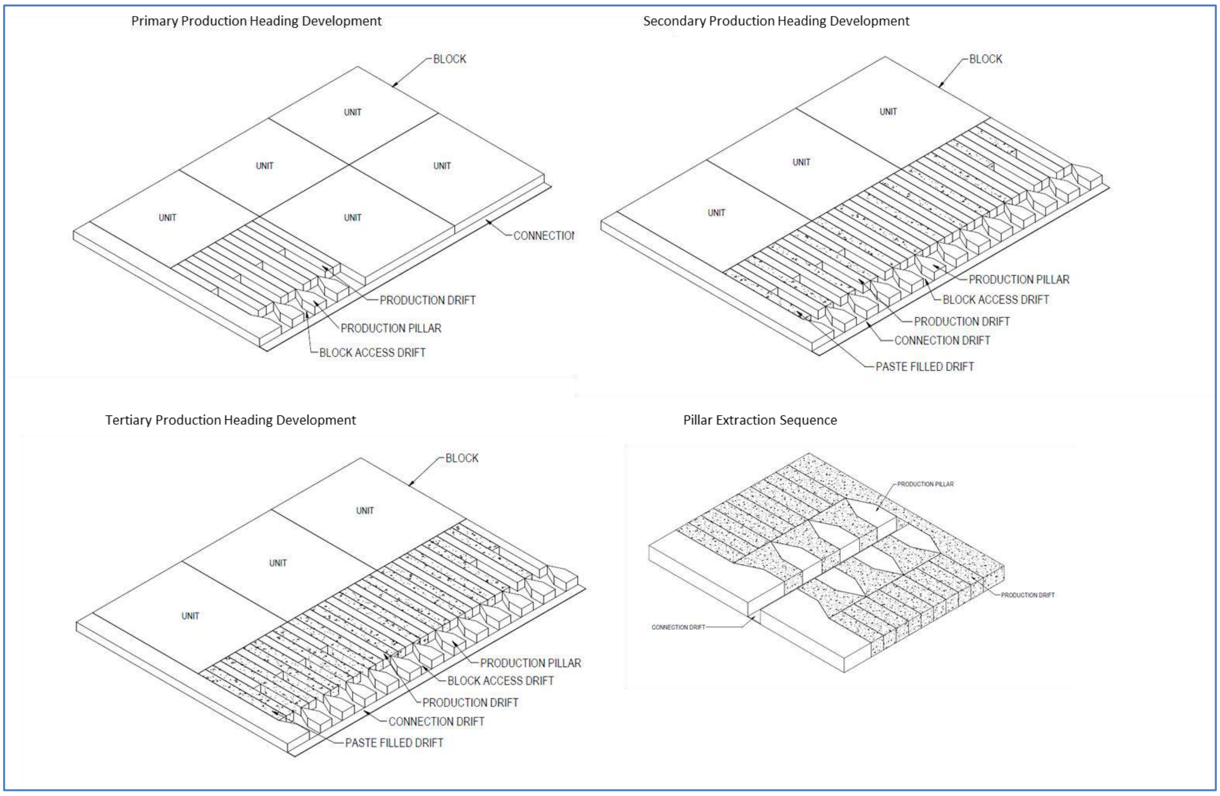


Figure by Stantec, 2019.

For ore thicknesses greater than 6 m, a second mining lift will be required. The second lift will be accessed from a new connection drift developed directly above the underlying connection drift. Based on the findings from the PFS geotechnical review, the block-and-unit configuration and extraction sequence for the second lift can be done identical to lower lift; however, further investigation is required to confirm the best layout for the second lift.

The first lift will be mined at 6 m high, and the remaining ore will be evaluated to determine its mineability. The second lift must meet the cut-off criteria using a minimum mining height of 3 m.

Mining Method Extraction

The extraction ratios for room-and-pillar mining are based on the dip, height, and depth of the panel and range from 55% to 70%.

The drift-and-fill method, including pillar extraction, assumes approximately 96% extraction of the mined shapes. The block sizes are based on using an 100% extraction of the ore in the first phase (primary, secondary, and tertiary) and 75% extraction from pillar extraction. Losses associated with mining and dips were applied using dilution and recovery factors.

Dilution

To obtain dilution grades, two dilution shells were constructed around each production shape to report the grade and density outside of the targeted resource, which include a 1 m hangingwall (HW1) dilution shell and a 1 m footwall (FW1) dilution shell. The block model interrogated the dilution shells by block center to provide the dilution grades and densities for each shell. The grades and densities were then applied to the volume of dilution for each production shape to obtain the diluted tonnes and grade. The grade of any paste fill dilution was zero for all metals.

All development headings outside the production areas will be 5.5 m W x 6 m H. Conveyor drifts will have a cross section 7 m W x 6 m H. These development headings will have a flat back with arched corners. Back and wall dilution is assumed to have an average overbreak of 0.1 m. No floor dilution has been assumed since any overbreak will be left for roadbed construction. Some of the development headings will include low-grade tonnes. The dilution grades applied to the low-grade development will be applied to the overall dilution tonnes.

The ore development consists of the connection drifts that are 6 m W x 6 m H, with a flat back and arched corners. The internal dilution was calculated using ratios of ore and waste from the footwall and the hangingwall. Where a second lift exists above the first lift, the back dilution for the first lift is assumed to be 0%. In a single-lift scenario or the second lift, a dilution of 2% was applied to the hangingwall.

Room-and-pillar mining includes production only from the main and cross-cut drifts. There is no pillar extraction with this method. All drifts will be 6 m wide, and the height will be the ore thickness (minimum 3 m and maximum 6 m). For hangingwall dilution, overbreak is assumed to average 0.15 m. No dilution from the walls was considered since the pillar width must be maintained. Controlled blasting practices will be required to ensure that the walls are broken to design width and the flat or shanty backs are maintained.

The footwall dilution is a planned dilution and is based on the dip and thickness of the production panel shape. For thicknesses where the short side wall is less than 2.5 m high, the angle of the back is adjusted. The result is an increased hangingwall dilution. This occurs for all 3 m high thicknesses with a dip greater than 12° and all 3.5 m high thicknesses with a dip greater than 16°.

Drift-and-fill will be the primary method of ore extraction. The first pass will include the connection drifts and the primary, secondary, and tertiary headings in the production blocks. The second pass will be pillar extraction. In areas where a second lift of mining exists, the layout will mirror the first lift. All headings will be 6 m in width.

Primary drifts will have only hangingwall dilution since any overbreak into the ribs will be within the planned excavation. Secondary drifts will have hangingwall dilution and paste fill dilution from one rib. Overbreak in the rock rib will be in ore and will not count as dilution. Tertiary drifts will have hangingwall dilution and paste fill dilution from both ribs.

Areas that require a second lift will have overbreak in the hangingwall that will not be considered dilution for the first lift of development. Wall dilution will be the same as identified for single-lift production.

The second lift of a two-lift production will include hangingwall dilution and paste floor dilution. The floor dilution is assumed at 0.15 m average thickness and is a result of mucking.

The dilution percentages will vary based on ore dip and thickness as well as the extraction sequence. Primary headings will have no dilution in the walls. Secondary headings will have a single paste wall and an ore wall. Tertiary headings will have paste on both walls, as it is mining between a primary and secondary heading that has been filled. The total dilution is the sum of the three headings divided by three to get the average heading dilution. Single-lift dilution typically ranges from 5% to 20%, first-lift dilution typically ranges from 2% to 5%, and second-lift dilution typically ranges from 5% to 20%.

Pillar extraction dilution factors will be the same as those for the drift-and-fill production headings. The wall dilution for Phase 1 and the slash wall dilution will be 0.15 m. The same logic and drift-and-fill production factors will apply if there is a second lift.

Recovery Factors

The mining recovery includes allowances for equipment limitations, heading shapes, heading strike and dip angles, ore re-handling, and operator skill.

For primary development the recovery is 98.9%. Lost tonnage is a result of losses due to the corners of the drift and muck that settle into floor irregularities; 0.1 m of rock material is estimated to be lost on the floor.

Room-and-pillar and drift-and-fill mining are development-intensive and will have recoveries similar to ore development. Some material will be left along the corners of the walls but will be recovered during the pillar extraction phase, so a recovery of 98% is expected when the sill is flat. As the dip increases and the operating floor is kept flat, additional ore will be unrecoverable in the footwall wedge. For drift-and-fill mining of the single lift and first lift of a double-lift scenario and R&P, the recoveries will be the same. For the second lift of a double-lift, an average of 0.1 m will be excavated out of the paste-filled floor. This will result in a recovery of 100%, since all ore will be taken with the backfill dilution.

The production pillars recovery was reduced to 75% of the recovery values calculated. The reduced recovery was applied due to the risk and potentially unknown conditions of the area when extracting the remaining ore. Further evaluation will be required to validate this recovery.

Backfill

The following main backfill products will be used.

- Cemented Sand Fill (CSF).
- Cemented Paste Fill (CPF).

CSF will be used for approximately 10 months until the mill is producing sufficient tailings to supply CPF. Local surface sand will be mined and processed through the paste plant as CSF until the transition to CPF in October 2021. Once this occurs, CPF will become the primary backfill method.

Based on an evaluation of different types of backfill characteristics (e.g., strength, quantity), it was concluded that paste backfill is the viable option for the Kakula Project. Paste backfill would serve the LOM backfilling requirements and minimize the quantity of tailings reporting to the Tailing Disposal Facility (TDF). Paste backfill will only be introduced in the drift-and-fill mining areas; the initial room-and-pillar mining areas will not be backfilled. Various optimisations have been identified during the study; the most notable are:

- The need to cyclone the tailings due to the high ultra fines content of the concentrator tailings.
- The opportunity to reduce high volumes of water being transferred to and from the backfill plant by locating the cyclones in the concentrator.
- The option of sand addition to reduce the tailings required as well as reduce the cement content.
- The possibility of splitting the paste plant into two sub-plants to reduce the number of surface booster stations. This would require a second plant infrastructure—the equipment required would remain the same but with a reduction in the number of booster stations.

These optimisations will be implemented in the next phase of the Project.

The testing concluded that the tailings expected to be produced by the Kakula concentrator could be used in the formulation of paste backfill; however, they had a very fine particle size distribution (PSD), and the tailings filtered poorly due to the presence of elevated concentrations of mica mineralization. The mica mineralization also required elevated addition levels of normal Portland cement to attain the target backfill UCS.

It was decided to reduce the fines content of the tailings through conditioning by hydrocyclone (cyclone) separation. The underflow of the cyclone separation process provided a coarser PSD that proved to be better suited to vacuum disc filtration.

The target UCS required for the fill was given as a minimum of 90 KPa as provided by SRK in the geotechnical report. The UCS testing was undertaken with paste at 178 mm (7 inches) slump. For a target UCS of 90 kPa a 5% binder addition will suffice for the 254 mm (10 inches) slump, as interpreted from the results obtained in the UCS testing. A 4% addition of binder addition rate provided a UCS of 126 kPa, and a 6% binder addition rate provided a UCS of 355 kPa, both at 28 days curing.

Table 16.10 and Figure 16.16 detail the fill requirements by year.

Table 16.10 Fill Requirements by Year (tonnes x 1,000)

Backfill Method	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**
CSF	–	5.2	484.8	–	–	–	–
CPF	–	–	255.3	1,517.1	1,956.40	2,873.0	3,242.1
Total Backfill	–	**5.2**	**740.1**	**1,517.1**	**1,956.40**	**2,873.0**	**3,242.1**
Backfill Method	**2026**	**2027**	**2028**	**2029**	**2030**	**2031**	**2032**
CPF	3,424.10	3,398.5	3,389.80	4,022.7	3,851.5	4,137.4	3,898.5
Backfill Method	**2033**	**2034**	**2035**	**2036**	**2037**	**2038**	**2039**
CPF	3,954.60	3,575.2	3,856.9	3,813.1	3,607.5	3,993.10	3,701.6
Backfill Method	**2040**	**2041**	**2042**	**2043**	**2044**	**2045**	
CPF	3,701.0	2,305.7	1,463.0	1,225.7	1,113.5	802.8	

Figure 16.16 Backfill Schedule

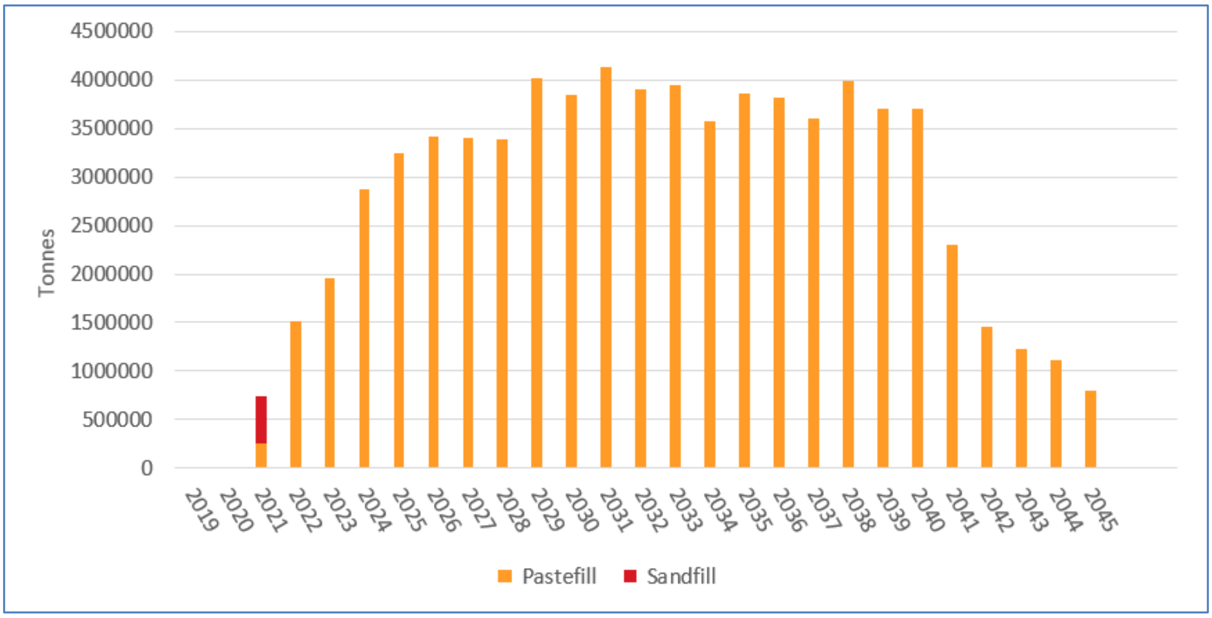


Figure by Stantec, 2019.

The underground paste distribution system is based on the LOM extents and requirements of the drift-and-fill mining. Based on the mine plan, the extents of the distribution system required a paste slump of 254 mm (10 inches) in order to deliver paste to the extents of the mine using reasonable pumping pressures.

Flow models and distribution system pressure profiles were completed for the paste distribution system required for the LOM extension of the underground distribution system (UDS). Based on the results of the flow models it is estimated that positive displacement pumps will be required over the LOM, allowing 254 mm (10 inches) slump paste to be delivered to the majority of the areas underground. Five surface booster stations are required in the current design to provide paste to the extents of the mine. The flow models indicate that underground booster stations will not be required.

The pipeline friction losses have been estimated and should be verified through flow loop testing, and flow models adjusted accordingly, in the feasibility phase of the Project.

To meet the backfill demand and maintain a suitable pipeline velocity between 1 and 1.5 m/sec, the UDS will comprise a 254 mm (10 inch) diameter borehole and pipeline system.

The surface boreholes are fitted with a 203 mm (8 inch) casing to mitigate the ingress of water into the paste, which would detrimentally effect paste quality. The volume of paste delivered through each borehole does not justify the use of any special liners unless geotechnical findings dictate.

Due to the length of the paste's transit time in the distribution system, admixtures will likely be required to retard the binder hydration process while the paste is in transit. Lab testing will be required to determine the suitable admixtures.

The following risks have been mitigated to acceptable levels through the duration of the study:

- Paste backfill availability when process plant stops during operation.
  - The paste plant has the capability to operate at utilization rates above that normally required to meet the needs of mine production to enable catch-up capabilities when the concentrator continues to operate.
- The availability of binder used in the paste preparation process through discussions with cement suppliers to ensure that supply will be available.
- The possibility of processing all the available tailings through the cyclones and then storing a small quantity for use when the concentrator is not producing tailings has been included.
- Additional binder storage capacity in a dedicated building has been included to ensure that at least a two-week supply is available for use. The ultimate binder supplier has not been selected at this stage of study. This will be reviewed during the feasibility stage.

Mine Access Designs

There are two open-concept box-cut designs for the Study. The northern box-cut, which was completed prior to the Study, incorporates two portals. The southern box-cut, which is planned to be completed by 1 April 2019, will have a single portal.

The main decline and conveyor decline were in development prior to the start of the Study. The main conveyor decline will be wide enough to accommodate the conveyor as well as large mobile equipment, such as an LHD or truck, while still maintaining pedestrian access. Travel direction in the declines is anticipated to be uphill in the main conveyor decline and downhill in the main service decline. Approximately 1,030 m from the portal of the declines, twin access drifts will be developed off the main service decline to the perimeter drifts. These drifts will also provide access to the main shop and the initial truck tip area. (See Figure 16.17).

Figure 16.17 Main Access Development

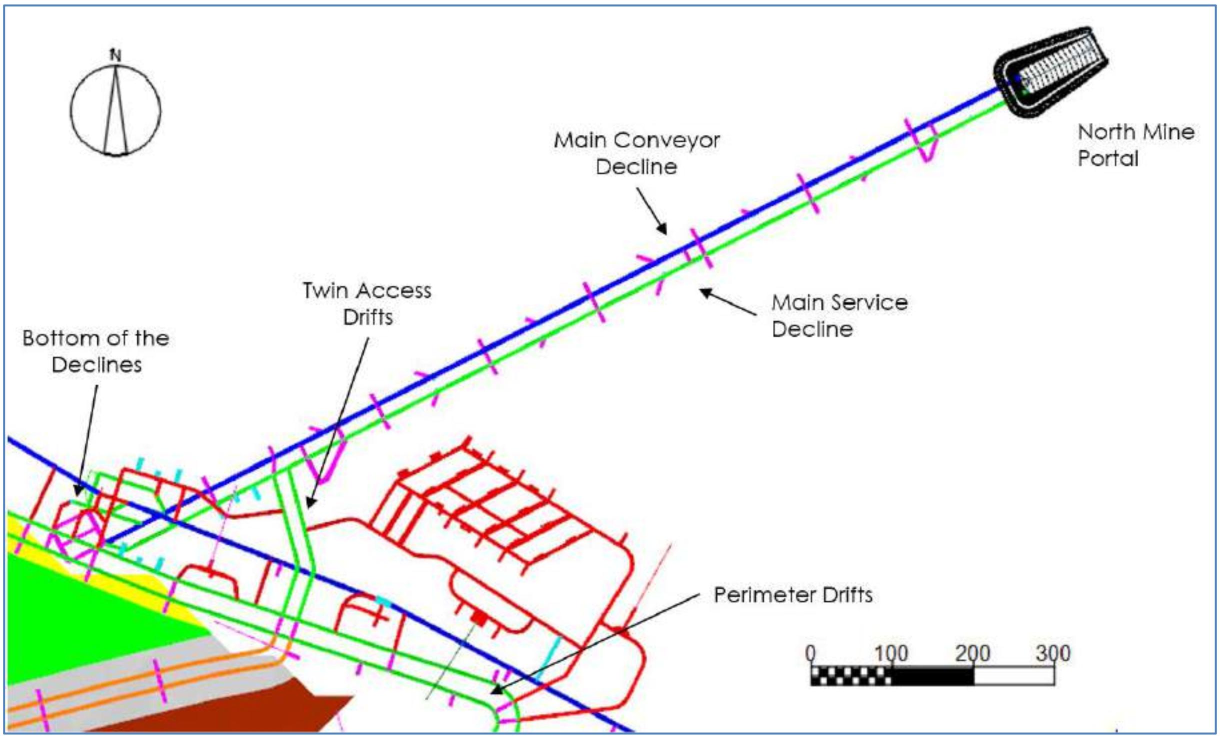


Figure by Stantec, 2019.

The south decline is a 5.5 m W x 6 m H single decline from surface to the south perimeter drift. It will target the perimeter drift elevation that will provide access to the south-east and south-west dual perimeter drifts (Figure 16.18).

Figure 16.18 South Access Development

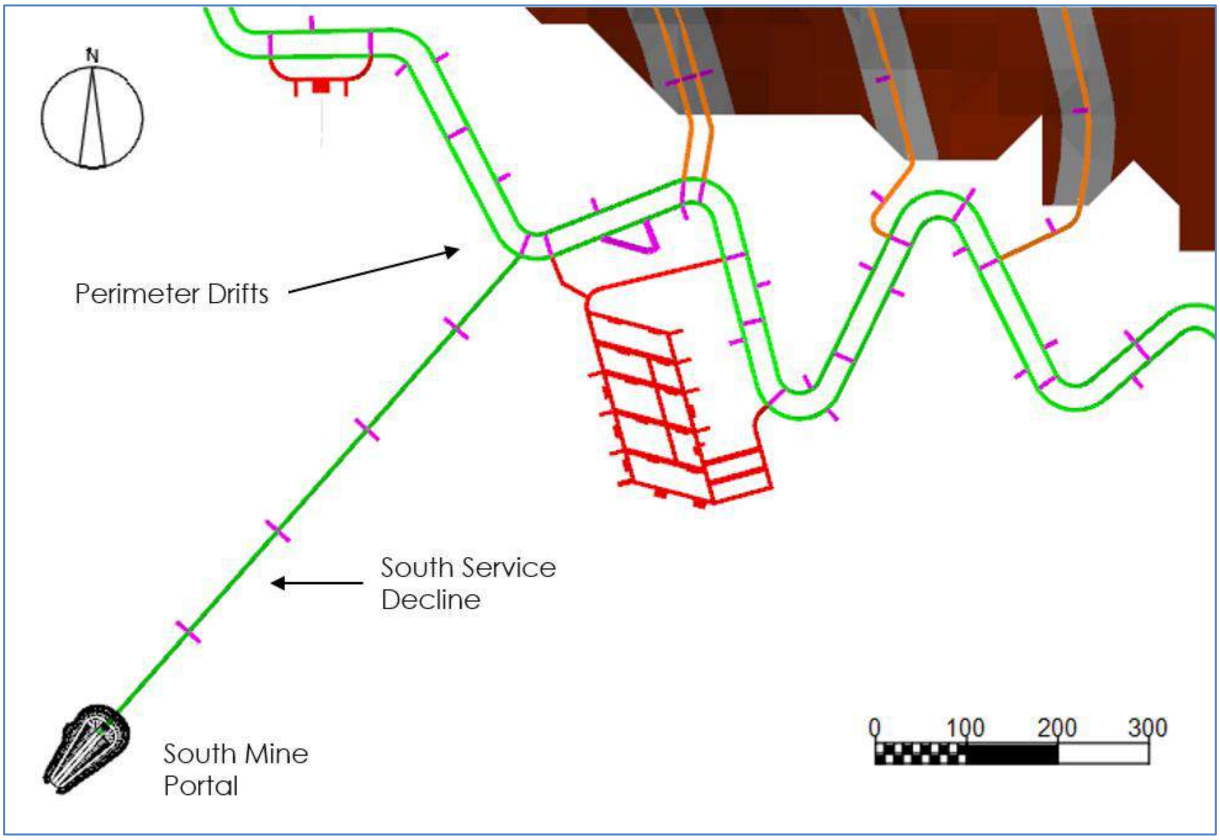


Figure by Stantec, 2019.

From the bottom of the north and south declines, a pair of 5.5 m W x 6 m H perimeter service drifts will be driven to the east and west extremities of the deposit and will serve as the primary accesses to the production areas and underground infrastructure. These drifts will serve as the primary intake and exhaust ventilation circuits and will connect with a series of intake and exhaust ventilation shafts.

The perimeter conveyor drifts are located outside the north perimeter drifts and will terminate at the main conveyor decline.

Connection drifts between the north and south perimeter drifts will provide access and ventilation to the planned mining areas.

The maximum gradient for this development will be 8.5°.

Mining Areas

For drift-and-fill mining, connection drifts will be developed between the north and south perimeter drifts. These will serve as the main accesses to the production blocks. The spacing between connection drifts (center to center) is dependent on the depth and dip of the orebody (Table 16.11).

Table 16.11 Connection Drift Spacing

Depth (m)	Plan Pillar Length (m)					
	0°–12°	12°–16°	16°–20°	20°–25°*	25°–30°*	30°–35°*
0–400	204.4	207.9	212.1	209.0	210.4	213.2
400–600	248.9	252.9	257.7	254.1	255.8	259.1
600–800	293.3	297.9	303.3	299.3	301.2	304.9
800–1,000	337.8	342.8	349.0	344.4	346.6	350.7

Based on 85% mining block extraction of initial pass mining.

Mining blocks will be divided in the connections drifts and be extracted based on the grade and geotechnical requirements.

The initial mining panel is located south of the main decline will be room-and-pillar. The center connection drifts between the north and south declines will serve as the primary access to the room-and-pillar mining area along with the adjacent perimeter drifts. Room-and-pillar mining will progress from the center connection drift to the west.

16.2.1.3 Mine Development and Production Schedules

The development schedule focuses on establishing necessary mine services and support infrastructure to set up the initial production mining areas and to ramp up to 6 Mtpa ore production. Based on a 360-day operating schedule, the production goal is to sustain full production for 15 years.

Mine development and production will occur in the following phases.

- Phase 1 – Development of the north declines (early 2019).
- Phase 2 – Development of the south decline and production rock handling facilities (2019).
- Phase 3 – Start room-and-pillar mining (February 2020).
- Phase 4 – Development of initial double connection drifts to establish primary ventilation circuit and start of drift-and-fill mining (Oct 2020).
- Phase 5 – Initial full production (2024).
- Phase 6 – Life-of-Mine.

Table 16.12 summarises the LOM development and production results.

Table 16.12 Life-of-Mine Development and Production Summary

Waste Development	
Lateral (m)	99,966
Lateral (t)	9,580,395
Mass Excavation Lateral Equivalent (m)	3,521
Mass Excavation (t)	351,277
Vertical (m) [1]	38,215
Vertical (t) [1]	431,901
Low-grade (m) [2]	2,971
Low-grade (t) [2]	254,642
Total Waste Development	
Total (m)	144,674
Total (t)	10,618,214
Production by Mining Method	
Ore Development (m)	37,430
Ore Development (t)	3,874,189
Room-and-Pillar (m)	35,518
Room-and-Pillar (t)	1,975,883
Drift-and-Fill (m)	1,485,944
Drift-and-Fill (t)	113,877,557
Total Ore Production [3]	
Total Development (m)	37,430
Total Production (m)	1,521,462
Total Tonnes (t)	119,727,630
Diluted Grade	
NSR ($/t)	238.44
TCu (%)	5.48
S (%)	1.51
As (%)	0.00
Fe (%)	4.83
Density (t/m^3)	2.83

1. Vertical development includes boreholes.
2. Low-grade cut-off NSR grade – US$60.
3. Ore cut-off NSR grade – US$80.

The following conditions were used in developing the LOM schedule.

- Proximity to the Main Accesses and Early Development.
- High NSR and Tonnage.
- Ventilation Constraints.
- Mining Sequence Constraints.
- Rock Mechanics Constraints.
- Backfill Constraints.

Using the above strategy, appropriate mining blocks were targeted and scheduled to achieve the highest possible grade profile during ramp-up and full production. Low-grade material was classified by NSR values, which range from $60 to $80.

Mine Development Plan and Scheduling

Productivity Rates

The effective operating hours per shift represent the time a crew spends in their working place performing work. This was estimated to be 8 h/shift. The effective working time per shift was applied throughout the zero-based rate calculations except for borehole installation, raiseboring, and underground facility construction, which were Contractor-supplied based on their own daily shift schedules and included in the schedules and productivity estimates for these activities.

For primary development, the rates were calculated using first principles. Cycle inputs were obtained from various sources (e.g., OEM, external consultants, specialists, historical files) and compared with inputs. The cycles were updated accordingly following team discussions. Mine productivities and schedule are based on the operating parameters shown in Table 16.13.

Table 16.13 Development Rates

Description*	**Single-Heading Performance (m/d)**	**Double-Heading Performance (m/d) (1)**
Perimeter Service Drift 5.5 W x 6 H – Semi-Arch	4.11	5.55
Perimeter Conveyor Drift 7 W x 6 H – Semi-Arch	4.10	5.53
Connection Drift 6 W x 6 H – Semi-Arch	3.73	5.03
Mass Excavation	2.98	N/A

(1) Double heading rates were used for re-muck bays, dams, and other bay development from drifts where it was appropriate.

Raiseboring rates provided by DRA used in the Project schedule are from Contractor experience or from recent Contractor quotations for the Study. Boreholes are raisebore pilot holes. All ventilation shafts and raises are assumed to be raisebored and include allowances for ground support.

The individual development rates for primary, secondary, and tertiary drift-and-fill were combined with paste filling and end-of-shift blasting restrictions in a block configuration and analysed to determine the net block production rate (t/d), which was used in developing the production schedule. For a block design with a maximum of 4 primary, 4 secondary, and 4 tertiary drifts, combined with end-of-shift blasting, 4 rounds/day/block was the maximum sustainable production performance.

The backfill type to be used in the drift-and-fill mining method is paste fill. The paste fill portion of the drift-and-fill mining cycle consists of the following three events.

- Preparation and fill fence construction at a 1 d duration.
- Fill time at 160 t/hr rounded up to the nearest day.
- Cure time at 28 d.

The drift-and-fill production block dimensions have been developed so that the excavation time would be greater than or equal to the paste fill time, and thus the paste fill time would be off the critical path of the block production. Golder provided the productivity estimates for the paste filling process.

The production rates were de-rated by a 26-of-31-day cycle to account for lost time during the production cycle. Additionally, second lift production was reduced to 75% of the first lift performance. Difficulties due to factors such as rock bolts in the muck piles and working on paste backfill will have a significant effect on the rates. See Table 16.14.

Table 16.14 Production Rates (Drift-and-Fill)

		Production Rates (t/s)			
		Mining Blocks and First Pass of Production Pillars		Pillar Extraction	
		1st Lift	2nd Lift	1st Lift	2nd Lift
Average Height	3 m	775	600	200	150
	4 m	950	750	250	175
	5 m	1,175	900	300	225
	6 m	1,375	1,075	350	275

The average cross section for room-and-pillar production drifts was used for the zero-base estimates of the three SRK-defined ground conditions. The average ground support was used to determine which of the face drilling, LHD mucking, and ground support activities of the 3.1 m average center height room-and-pillar panel was the controlling critical activity. In room-and-pillar production panels, it is assumed that after a startup period to create seven to eight faces, there will always be a workplace for the critical activity to occur. The production rate of these panels will be determined by the rate of the critical activity. The rates estimated were 439 t/d during ramp-up and 1461 t/d during full production.

Development Schedules

The preproduction/ramp-up period is defined as starting on 01 January 2019 and ending Q1 2021. Key development activities to be completed in this period include the following:

- South decline.
- Initial ventilation raises VS1 and VS2.
- Materials handling infrastructure around bins and conveyor loading area.
- Center connection drifts between north perimeter service drift to south perimeter service drift.
- Construction and development of other critical infrastructure (e.g., conveyors, initial truck tip points, return water pumping, refuge stations, underground workshops.)

The ramp-up period to full production concludes at the end of 2023. Figure 16.19 illustrates the development metres associated with the preproduction and ramp-up activities.

Figure 16.19 Preproduction/Ramp-Up Development Schedule

Figure by Stantec, 2019.

The development schedule beyond the initial preproduction period and ramp-up targets the areas required after 2023 to support the LOM plan. This would include excavating the perimeter service and conveyor drifts ahead of production mining blocks, access to and construction of the remaining ventilation shafts, and the construction of the required materials handling and support infrastructure. Figure 16.20 illustrates the development metres associated with the LOM activities.

Figure 16.20 Life-of-Mine Development Schedule

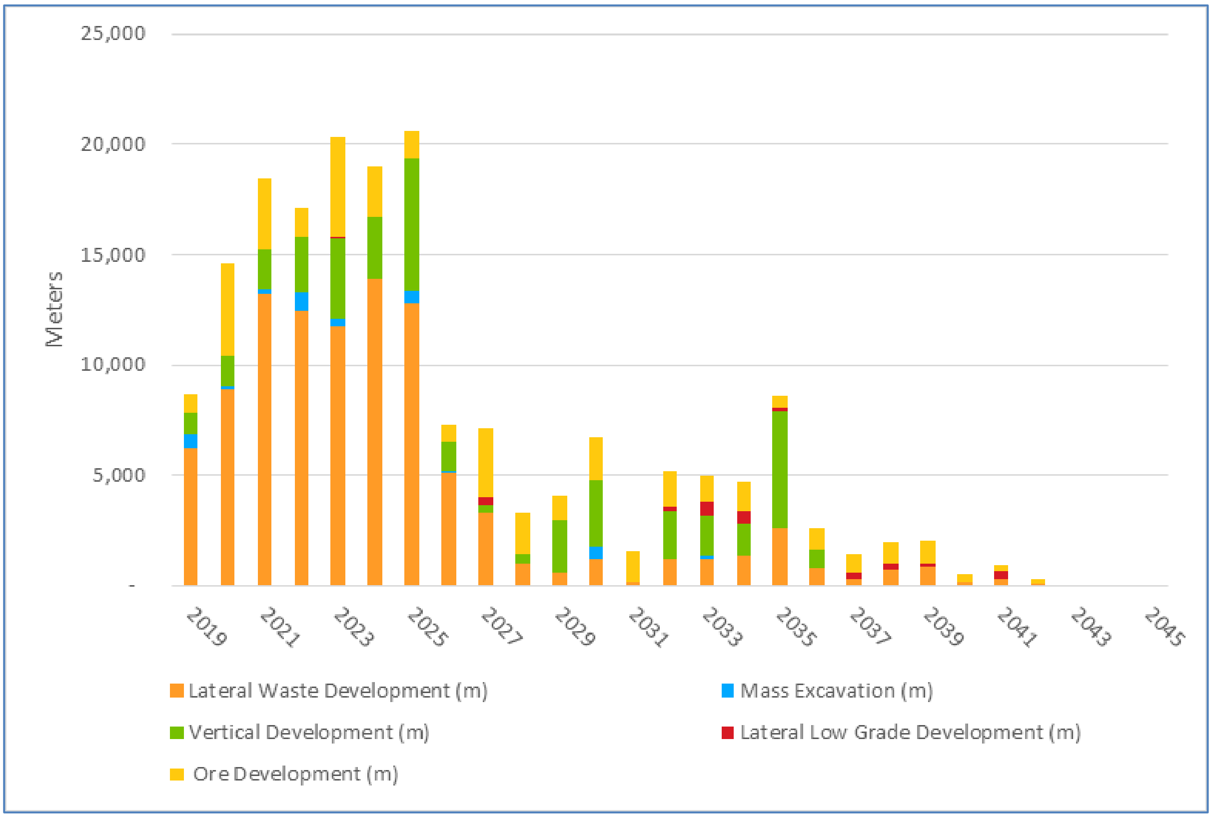


Figure by Stantec, 2019.

Mine Production Plan and Scheduling

Table 16.15 presents the annual preproduction/ramp-up targets and resulting scheduled tonnes that were set to meet the 6 Mtpa production rate.

Table 16.15 Preproduction/Ramp-Up Targeted and Scheduled Tonnage

Production Schedule	Years	Targeted Tonnes	Scheduled Tonnes
Initial Production Mining (2019)	1	200,000	94,188
Ramp-Up (2020)	1	1,200,000	889,329
Ramp-Up (2021)	1	2,000,000	1,931,422
Ramp-Up (2022)	1	3,000,000	3,171,726
Ramp-Up (2023)	1	4,500,000	4,690,317
Initial and Ramp-Up Total	5	10,900,000	10,776,982

Figure 16.21 presents the resultant scheduled preproduction/ramp-up tonnes and NSR.

Figure 16.21 Preproduction/Ramp-Up Tonnes and Net Smelter Return

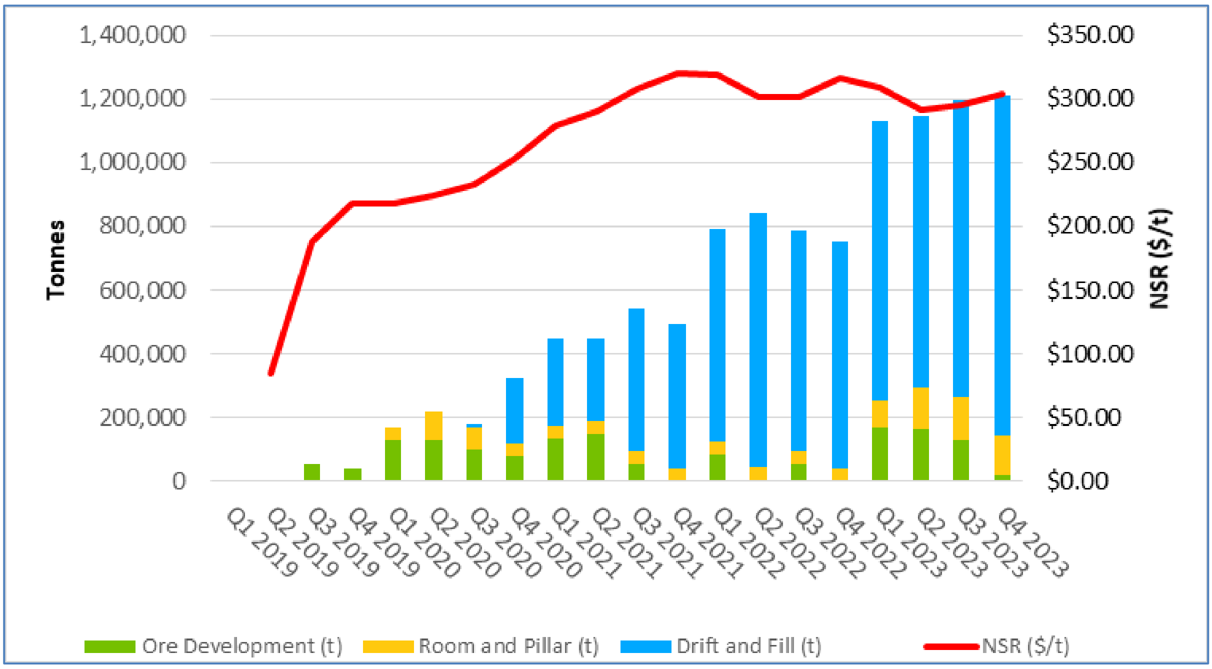


Figure by Stantec, 2019.

Full production of 6 Mtpa is sustained for 15 years, starting in 2024 and tapering off after 2038 as the reserve is depleted. The mining blocks were scheduled so that a higher NSR value is achieved early in the mine life. Figure 16.22 illustrates the LOM production schedule and NSR.

Figure 16.22 Life-of-Mine Production Schedule and Net Smelter Return

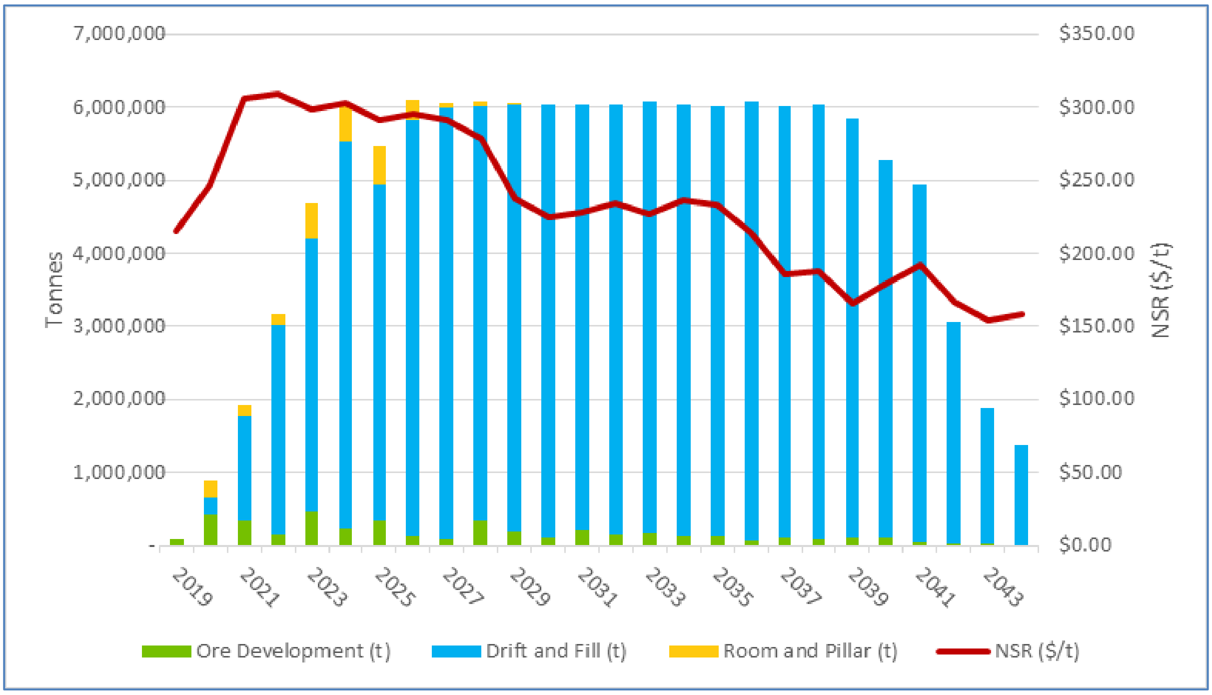


Figure by Stantec, 2019.

16.2.1.4 Underground Infrastructure

Mine Ventilation System

The underground mobile equipment fleet will be diesel powered, and mine air cooling will be implemented as required to maintain underground working air quality within the appropriate limits. As a result, the mine ventilation system has been primarily designed as a "pull" system, with surface fans on the exhaust shafts to improve control on the ventilation system.

The mine layout schematic illustrating the location of the ventilation shafts is shown in Figure 16.23. The ventilation system has been designed to provide fresh air through shafts on the north side, and the north and south declines, while exhausting return air through the south side ventilation shafts. The fans are located at the exhaust shafts on surface, where possible, to reduce heat gain in the fresh air supply and better control the airflow through minimizing leakage. Also, all the main fans will be equipped with variable frequency drives, to provide the ability to modulate the airflow being exhausted from each shaft. South African regulations for mine ventilation and industry best practices were considered in assessing the ventilation requirements.

Figure 16.23 Plan View of Ventilation Shaft Locations

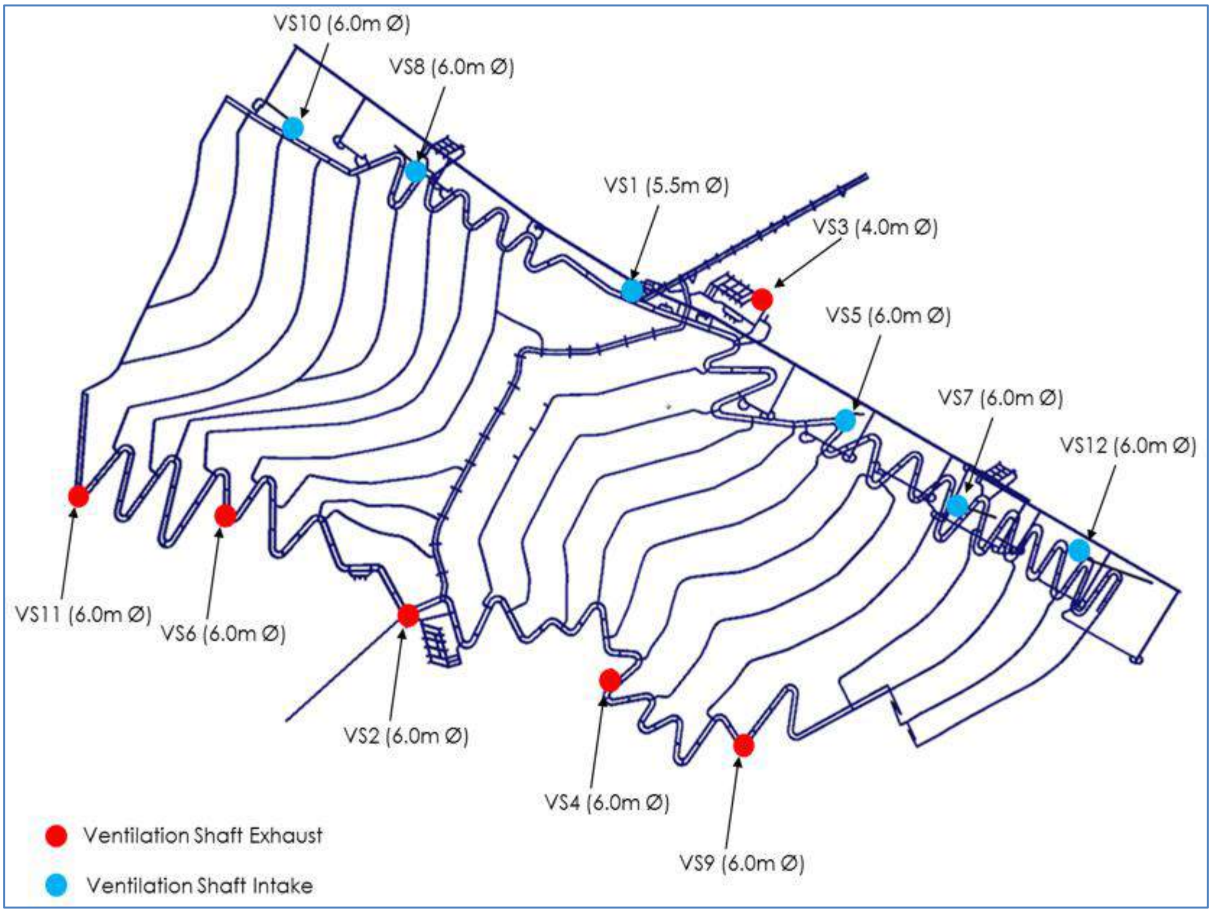


Figure by Stantec, 2019.

Underground booster fans will be installed in the west and east workshops to help ventilate the shops and maintain the conveyor drifts under negative pressure. A summary of the primary and secondary ventilation fans is provided in Table 16.16.

Table 16.16 Primary and Secondary Ventilation Fan Supply

Shaft/Raise Name	Type	No. of Fans	Fan Arrangement	Date Required
VS1	Intake	2*	Parallel	May 2019
VS2	Exhaust	2	Parallel	Mar 2020
VS3	Exhaust	2	Single	Dec 2019
VS4	Exhaust	2	Parallel	Jul 2021
VS6	Exhaust	2	Parallel	Aug 2025
VS9	Exhaust	2	Parallel	Mar 2024
VS11	Exhaust	2	Parallel	Mar 2031
UG Workshop – West	Exhaust	1	Single	Jun 2026
UG Workshop – East	Exhaust	1	Single	Mar 2025
East UG Booster 1	Exhaust	2	Series	Jan 2035
East UG Booster 2	Exhaust	2	Series	Jan 2035

*Only during the preproduction period.

The airflow requirement calculations take into consideration the utilization factor of the mobile equipment and is rated at 0.063 m^3/s per kW. The equipment shows the crew requirements for development, production, and rock haulage k (ore or waste). Including a system leakage factor of 15%, a maximum total flow of approximately 2,000 m^3/s will be needed at full production.

The total flow requirement is for the maximum equipment requirement, with 12 drift-and-fill crews (one crew assigned to each block), 1 room-and-pillar crew, and 10 development crews. Full production is reached earlier in the LOM, with the airflow requirement adjusted according to equipment usage.

Cooling System Description

BBE was subcontracted by Stantec to design the surface cooling and refrigeration systems for production build-up and permanent mining phases.

There are four discrete 5 MWR air cooling systems located on surface at the top of shafts VS1, VS5, VS7 and VS8. The coolers will be phased in over the LOM, with maximum installed cooling of 20 MWR in 2033. The four installations will be identical, comprising bulk air cooler (BAC), refrigeration module, plant building, condenser cooling towers, water pump systems, and electrical and control systems.

Each 5 MWR cooling system will be made up of the following three main components:

- BAC.
- Refrigeration Plant Room.
- Heat Rejection Facility.

Underground Dewatering

The underground dewatering system consists of a series of collection dams and cascade transfer dams on the north and south perimeter drifts, which will pump to underground and surface settler systems. An overview of the underground dewatering system is illustrated in Figure 16.24.

Figure 16.24 Underground Dewatering System

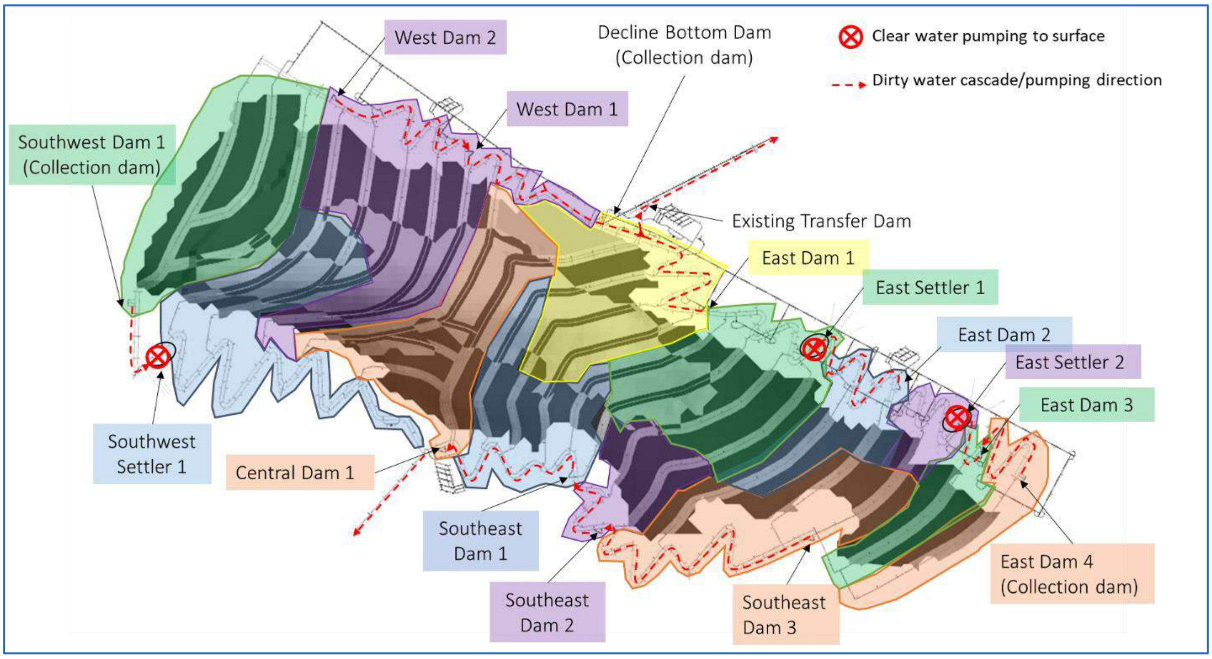


Figure by Stantec, 2019.

The collection dams will be equipped with 90 kW submersible pumps capable of delivering approximately 100 L/s, with a total head of approximately 57 m (vertical head 52 m). The dams will be equipped with a primary and secondary de-gritting wall. The de-gritting walls will allow water and fine silt to overflow into the dam through positively angled holes and an overflow arrangement.

Transfer Dams

Transfer dams will be equipped with 1 to 4 pump trains. Each pump train will comprise three 220 kW single-stage dirty water centrifugal pumps placed in series. Each pump train will be capable of delivering 200 L/s at a total head of approximately 200 m (vertical head 180 m).

The dams will be equipped with a primary and a secondary de-gritting wall. The de-gritting walls will allow water and fine silt to overflow into the dam through positively angled holes and an overflow arrangement.

The mine dewatering design includes surface and underground settlers.

Inflow Rates and Basis of Designs

Inflow rates were estimated by Golder. These inflows were estimated for the mining blocks that would be open for mining and backfill operations for each connection drift and summarised for maximum expected inflows to each of the designed dams. Inflows range from 12 L/s to 109 L/s depending on the area and catchment.

Fissure water will be encountered in varying amounts; this will be a function of the surface area of the block being mined and a known flow rate over a known area. The fissure water will mix with mine service water, seepage from paste fill, and paste fill line flushing. The dirty water mixture will be collected at each block in a floor sump. Submersible pumps will then be used to move the dirty water to a pump box. From the pump box, the dirty water will either be sent to a north or south perimeter sump or dam.

Face pumps will be used for dewatering during heading advance and any time an active mining face is producing water. A face pump will pump to a portable skid-mounted tank pumping station of which will report the water to a perimeter dam.

The storm water return system will be an independent return water system, specifically installed to cater for protection against storm water ingress into the declines via the north portals. The system will comprise the following two pumping systems:

- Box-cut Dewatering Sump and Pumps.
- 150 m Transfer Dam and Pump Station.

The box-cut dewatering system will comprise a sump, 3 x 90 kW submersible pumps, and a 37-kW submersible slurry pump. The sump will be positioned in the middle of the main decline box-cut. Drains will extend across the lowest point of the box-cuts, immediately in front of the portals. These drains will collect surface run-off water and deposit it into the sump for dewatering.

The system has been designed to cater for the following:

- Normal operating conditions of 30 L/s slurry reporting to the surface settlers.
- Heavy rainfall or flooding conditions of the 360 L/s water reporting to the surface settlers.

The storm water control system for the south decline will be the same arrangement.

Rock Handling System

The mining rock handling system will consist of several truck tips positioned along the northern edge of the ore deposit. Each truck tip will feed via apron feeder and sacrificial conveyor onto a series of transfer conveyors that will feed onto the east and west main conveyors. The conveyors will feed east and west crushers that will reduce the rock size and feed onto another perimeter conveyor, which convey the material into the transfer bins located at the bottom of the main decline. A single main decline conveyor will transport the ore or waste to surface from where it will be distributed to either the bulk stockpile or via the ROM stockpile to the plant.

The system can be separated into western, eastern, and main decline rock handling systems, as illustrated in Figure 16.25.

Figure 16.25 Underground Rock Handling System Overview

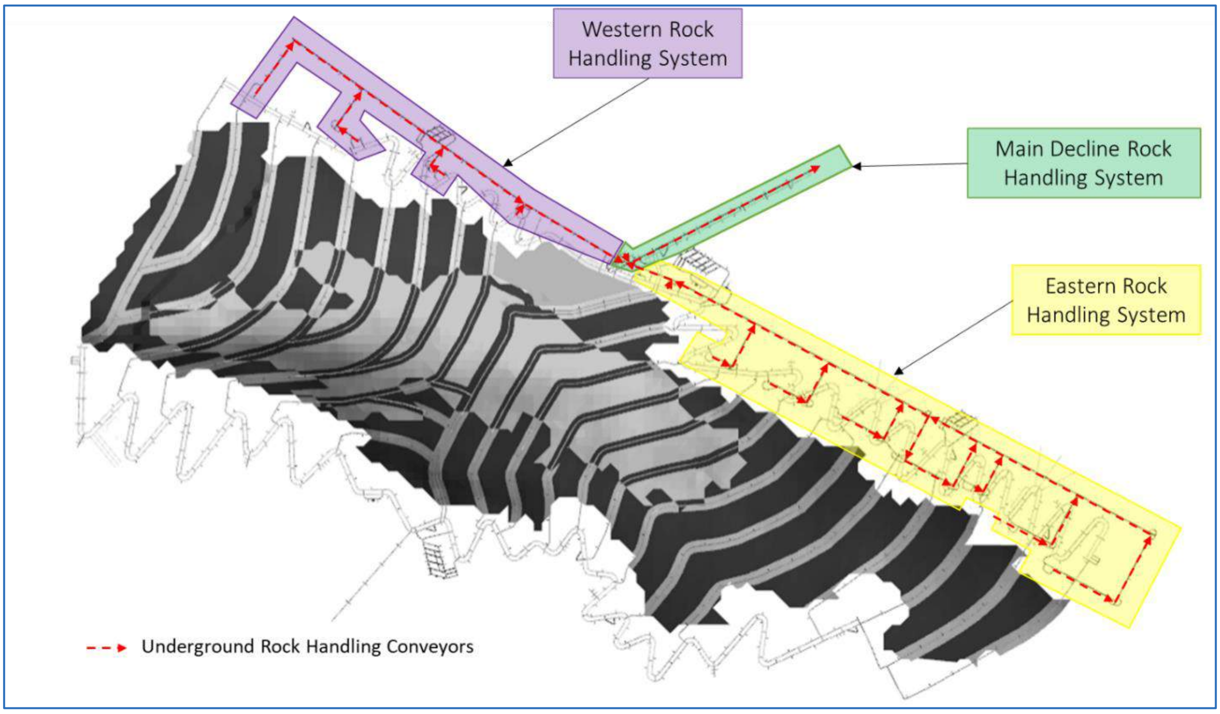


Figure by Stantec, 2019.

The western rock handling system comprises a total of four truck tips that feed ore and waste through a series of conveyor belts into the west crusher. Crushed rock will exit the crusher chamber and feed into the main decline's west transfer bin via the west transfer bin feed conveyor. The western rock handling system has been designed to deliver a total capacity of 2,000 t/h.

Ore or waste rock will then be fed by an apron feeder at the bottom of the transfer bins through a series of conveyor belts into the west crushing chamber. The crushed rock will be transported by the west transfer bin feed conveyor into the west transfer bin located at the bottom of the main decline.

The eastern rock handling system will comprise a total of 8 truck tips that will feed ore and waste through a series of conveyor belts into the east crusher. Crushed rock will exit the crusher chamber and feed into the main decline's east transfer bin via the east transfer bin feed conveyor. The eastern rock handling system has been designed to deliver a total capacity of 2,000 t/h. This includes the design capacity of the east main conveyors, east crusher chamber infrastructure, and the east transfer bin feed conveyor. This system will be fed via the 8 truck tips and associated apron feeders, sacrificial conveyors, and transfer conveyors. These components have each been designed to a 1,000 t/h feed rate.

Ore or waste rock will then be fed by an apron feeder at the bottom of the transfer bins through a series of conveyor belts into the east crushing chamber. The crushed rock will be transported by the east transfer bin feed conveyor into the east transfer bin located at the bottom of the main decline.

For both the western and eastern systems, rock will be loaded in the production areas using a combination of LHDs paired with underground haul trucks. The ore will then be truck-hauled along the perimeter drifts to the nearest planned truck tip on the northern perimeter of the orebody. Rock will pass through the static grizzly into a transfer bin. Oversize rock will be broken on the grizzly using a fixed hydraulic-boom rock breaker. Provisions will be made for tramp iron or scrap metal removal at each of the truck tips as well as a final removal point at the transfer from each of the sacrificial conveyors.

Main Decline Rock Handling System

The main decline rock handling system will receive feed from the east and west transfer bin feed conveyors as well as from 2 truck tips. The reception rate of waste or ore feed will be controlled and can be accepted from one, all, or any combination of the feed sources at a time. Rock from the three transfer bins will be controlled by apron feeders and fed through two parallel sacrificial conveyors onto the main decline conveyor belt, which will transport the rock to surface. The main decline rock handling system has been designed to deliver a total capacity of 2,000 t/h to surface. This includes the east and west apron feeders, main decline sacrificial conveyor, and main decline conveyor belt.

The west decline transfer bin will be equipped with a truck tip for early production and equipped the same as the other tips. The decline truck tip will be installed to cater for early development and to supplement early production. The designed feed rate of the decline truck tip is 800 t/h of grizzly-sized material. Similar to the west transfer bin truck tip, it will be equipped the same as all the other truck tips.

The main decline will be located centrally and to the highest elevation of the orebody, which is beneficial to the logistics of the underground ore and waste handling systems. The main decline conveyor system is planned to be installed as soon as practical once access is granted, to assist with the rock handling to surface and ease strain placed on hauling development waste through the main and service declines.

Mining Surface Rock Handling System

The mining surface rock handling system will receive ore and waste from underground via the main decline conveyor. The system has been designed for a 2,000 t/h capacity.

A series of conveyor belts will transport the rock directly or indirectly to the ROM stockpile, which will serve as the main buffer feed to the processing plant. The battery limit of the mining rock handling infrastructure is at the point of discharge of the ROM stockpile feed conveyor.

The system will allow for the storage of ore or the separation of waste at the bypass stockpiles, which will either be loaded out and stored at the bulk stockpile for later reclamation or dumped at the waste rock dump for reuse as crushed aggregate where required. The main decline conveyor will feed the ROM transfer conveyor that is installed with a diverter chute to split the flow to either the bypass stockpile or the ROM stockpile feed conveyor.

On the bypass circuit, the bypass stockpile feed conveyor will feed the reversible bypass stockpile conveyor that splits the material into two stockpiles. The two stockpiles will be loaded out by front-end loader and hauled with a surface dump truck to the bulk stockpile area. Reclaimed ore will be transported via dump truck from the bulk stockpile to a bulk reclaim tip bin, which will discharge material at a controlled feed rate back onto the ROM stockpile feed conveyor.

Provisions at the bulk reclaim tip will be made for dust suppression, oversize separation, and tramp iron removal.

Ore Passes

The rock handling systems will all be installed with an ore pass chute with spile bar gates to allow for controllable discharge to the apron feeders. Ore passes will be located at the east and west truck tips. Ore passes will also be installed on the provisional tipping point at the east and west secondary conveying/crushing area as a control measure to feed the crusher. Moreover, ore passes will be installed at the east and west decline transfer bins as a control measure to feed the main decline sacrificial conveyor. The provisional decline truck tip will also be installed with an ore pass to control feed to the decline truck tip sacrificial conveyor, which in turn will feed the main decline conveyor.

Crushing Chambers

Crushers will be located on the eastern and western rock handling systems and will receive all rock feed from the respective production sections. The crushers will provide appropriate sizing of rock for the effective handling on conveyor belts and chutes as well as the expected processing plant equipment feed acceptance size.

Jaw crushers have been selected due to ease of maintenance and reliability in comparison to other viable options. A vibrating grizzly feeder with a 220 mm aperture will be used to remove undersize from being fed into the crusher and will prevent overloading and excessive wear to the crusher liner plates. Each crusher will be provided with a bypass chute for production to continue at reduced rates, should there be unexpected and extended downtime on the equipment. Vibrating feeders will be used in the crusher chambers as effective feed rate controllers to and from the crusher equipment. Crusher chamber throughput rates expected are less than 250 t/h through the crusher and 1,750 t/h as undersize bypass, thus totaling the required 2,000 t/h through the rock handling system. Dust suppression will be provided at the feed and discharge of the crusher circuit to prevent excessive harmful dust from becoming airborne.

Fixed Facilities

Paste Fill

The mass balance is based on the design criteria for paste having a 178 mm (7 inch) slump and a 5 wt% binder addition rate. If the tailings were uncycloned, the cement content would rise to 7%.

As indicated in Figure 16.26, the paste plant process has been designed to prepare a backfill product made from cycloned mill tailings.

Figure 16.26 Simplified Process Flow Diagram of Paste Plant

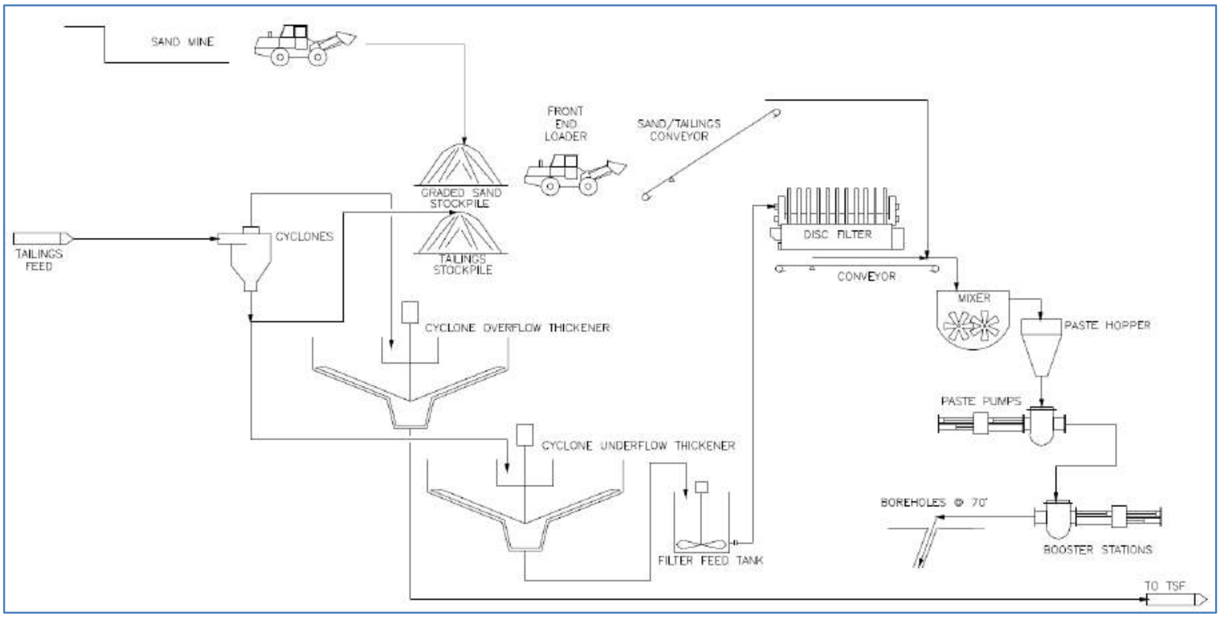


Figure by Golder, 2019.

The process is essentially comprised of agitated storage tanks, hydrocyclones, thickeners, vacuum disc filters, twin-shaft high-shear mixers, cement silos, and various process storage vessels. The process is designed to allow the best degree of process control that is needed to maintain consistent moisture content of the backfill for the delivery of paste into the stopes.

Tailings slurry will be delivered from the concentrator to the paste plant at approximately 55 wt% solids into an agitated receiving tank at the paste plant. When the paste plant is in operation, all the tailings generated by the concentrator will report to the paste plant. When the paste plant is not in operation, tailings from the concentrator will be cycloned and stockpiled for later use in the paste preparation process. The volume of the receiving tank offers some buffer capacity against tailings feed fluctuations resulting from the concentrator process. The size of the tank can be optimized in future design stages.

Sand will be used for paste fill for the first three quarters of the first year of mining operations where drift-and-fill mining is undertaken. Test work has been undertaken on three available sand deposits in the area, and the Kakula Dambo deposit was found to be the most suitable. Once the concentrator produces tailings, the sand will be stopped but will be available for mixing with the tailings to form a suitable paste.

The paste backfill will overflow out of the mixers to the paste hopper. The paste hoppers will provide surge storage to keep sufficient head over the paste piston pumps, allowing them to more readily achieve a high volumetric filling factor. The paste will then be pumped underground by the piston pumps through the appropriate borehole via surface booster stations.

The binder system consists of four 300 tonne silos. The cement will be metered into individual screw conveyors and then proportioned to the mixer contents via a loss-in-weight feeder. The loss-in-weight feeder will discharge into a final screw conveyor into the mixer.

Continuous monitoring of the process through a PLC system by on-stream instrumentation will allow the plant controller to adapt process flows and conditions in real-time, thereby maintaining quality control of the final product.

Compressed Air System

Production equipment will be equipped with on-board compressors and will not require additional compressed air for drilling purposes. Portable refuge stations will be self-contained, with no compressed air. Underground workshop office/lunchrooms will be equipped as permanent refuge stations and will be provided with compressed air. Other users of compressed air include the underground shops.

The compressors will be installed in a compressor house located on surface between the box-cut and the settling dams. Two 737 cfm compressors, one duty and one standby, will be installed and sufficiently sized to supply surface workshops, underground workshops, and 4 permanent refuge stations.

Fuel and Lubricant Distribution

The diesel distribution system will comprise a main surface storage facility, two vertical drop systems, and two underground facilities. The two vertical drop systems will be located at the UG Workshop – Central North and the UG Workshop Central South, each with an underground batching system. Fuel will be transported with UVs from the main surface storage facility to the east and west vertical drop facilities.

The main surface batching system will comprise a bulk surface storage facility with six 83 m^3 bulk storage tanks as well as receiving and dispensing pump systems.

Fuel distribution at the vertical drop systems will be via a pipeline, installed in a borehole, from the surface batching tanks to the underground batching tanks. Each underground batching system will consist of three 9 m^3 tanks. A fuel-dispensing system will provide fuel directly to the underground fleet and UVs.

Lubricants will be stored separately at the main surface storage facility. Lubes will be distributed underground via cassettes.

Waste fuel and lubes will be stored in tanks and transported to surface using cassettes to the appropriate handling facilities.

Water Systems

A surface potable water treatment plant and storage tank will feed a galvanised steel pipe that will be used to transport potable water underground via the main decline. Provisions have been made for a second line on the various pipe racks, should potable water requirements exceed the design values. Pressure-reducing valves (PRVs) will be provided to limit the pressure of the potable water lines to acceptable levels. The potable water line will provide clean drinking water as well as gland seal water to the underground pumping systems.

Service water will be obtained from the existing surface mine service water tank, which gravitates underground via a galvanised steel pipe. Service water consumption has been estimated based on 20 crews drilling using 3 L/s, for a total consumption of 60 L/s over the LOM.

Fire Protection System

The following two fire-water ring mains will be used in the mining area:

- Gravity-fed line in the main decline that services everything below a depth of 150 m.
- Pumped line for surface infrastructure and for underground protection up to a depth of 150 m down the main decline.

The ring main will supply the various systems throughout the area, including but not limited to the following:

- Strategically placed fire hydrants.
- Deluge water spray systems on conveyors.
- Deluge water spray systems on transformers, hydraulic power packs.

Fire hydrants will be installed along the entire length of each belt along with deluge systems comprising open nozzles directed at the main and return belts. The system will be operated and controlled by a normally closed 24 V DC solenoid.

Protection of all other areas underground will be achieved via portable/handheld equipment, with particular reference to the following areas:

- Workshops.
- Mini-Subs.
- Underground MCCs.
- Refueling Bays.

The equipment fleet (e.g., haul trucks, LHDs) will have built-in extinguisher systems along with hand-held units.

Substations on surface and underground will be protected by means of fire suppression systems utilizing heat and smoke detection combined with an automated extinguishing system.

Materials Handling Logistics

Delivery vehicles will report to the main gate and then be directed to the correct offloading area. Most materials will be delivered directly to the store areas by Suppliers. Materials will be delivered to, stored at, and distributed from one central main store on surface.

Through the materials control system, spares, consumables, or any other materials will be loaded into materials cassettes according to the area and section, ready to be delivered to underground sections. UVs will be used for materials transport. Similarly, for workshops, cassettes will be loaded on surface and unloaded at the place where the order originated.

Certain consumables (e.g., service kits, nuts and bolts, special oils) and spares (e.g., seal kits, O-rings, various cylinders, gear pumps) will be stored at underground laydown areas to prevent delays in production. “Typical” spares will be stored in a dedicated workshop store at the workshop area. Mining consumables such as roof bolts, resin, drill steel, etc., will be stored at dedicated mining storage areas.

Workshops

Mobile equipment will be supported by two surface workshops located in proximity to the main decline box-cut and four underground workshopsas well as temporary satellite workshops underground near the working face. Equipment wash bays will be located at the entrance of each workshop.

A heavy vehicle workshop will cater for all maintenance requirements of surface and underground production vehicles, as well as large mining support equipment. A light vehicle workshop will cater for all maintenance requirements of surface and underground LDVs, busses, and surface secondary support vehicles.

There are two centrally located underground workshops on the perimeter of the orebody, with one on the northern perimeter near the bottom of the main decline (the UG Workshop – Central North) and one on the southern perimeter near the bottom of the southern decline (the UG Workshop – Central South). The remaining two underground workshops will be located on the north-east and north-west perimeter drifts of the orebody (the UG Workshop – East and UG Workshop – West, respectively).

As the extents of the mine increases, there will be a need for a limited number of satellite workshops equipped to perform daily maintenance on slow-moving mobile equipment (e.g., jumbos, bolters). The satellite workshops will have limited capacity and will potentially be single-bay. Concrete floors, lighting, and a hoisting arrangement will be incorporated in their design.

The satellite shops will be in proximity to the major ventilation raises in the area. Furthermore, they will have the fire protection necessary for such a facility. These facilities will not have access to compressed air via the main distribution system and must produce compressed air locally.

Field service and repair/fuel and lube trucks will be employed to provide minor repair and lubrication services at the working face or other points of use.

Explosives Magazine

Surface delivery of Class 1 explosives will be to the surface explosives magazine via purpose-built explosives cassettes. Once the pallets are offloaded on surface, reloading for transport underground will start immediately.

The offloading facilities on surface will be fenced off with lockable gates, warning signs, lights, and fire extinguishers according to best practices and similar to other operating mines, while also adhering to the South African and DRC Explosives Act's requirements. As required by DRC mining regulations, guards will be positioned at the magazine.

There will be two magazines located underground one for detonators, one for package explosives ("sticks"), and one bulk emulsion.

Initially, the emulsion and sensitiser will be stored in tanks on surface located near the portal. Emulsion trucks will deliver emulsion every second day; however, sufficient storage capacity for four days will be required.

AEL provided the design and costs for a suitable vertical drop system. The following criteria was used for the design and estimate.

- Total storage capacity – 100 tonnes near the bottom of the main decline and 70 tonnes at the bottom of the southern decline.
- Bulk storage on surface.
- Vertical drop of 300 m or less.
- Bulk emulsion and sensitiser will be dropped in separate HDPE pipes through the same borehole.
- Emulsion and sensitiser will be transported using UVs to underground working areas.

The proposed system is well understood and in use at various other operations in Zambia. Emulsion cassettes will be used to store explosives near working areas or in the emulsion storage magazine.

Underground Office and Lunchroom

Provisions will be made for additional underground offices for underground production and workshop personnel. These offices will be constructed at the four underground workshops: UG Workshop – Central North, UG Workshop – Central South, UG Workshop – East, and UG Workshop - West.

Concrete and Shotcrete Facility and Distribution

A batch plant located on surface will be required to supply shotcrete and concrete to the underground workings for support and construction.

This plant has been designed to adjust the mix according to the requirements, with flushing facilities to clean out the complete system prior to any changeover. The batch plant will be equipped with a planetary mixer, capable of mixing a variety of concrete requirements, including steel and synthetic fibers. The mixer can produce high-quality shotcrete, ready-mix, mortar, and concrete. Production capacity will vary from 10 m^3/h up to 40 m^3/h, depending on the mixing period, materials used, and requirements.

Concrete will be mixed into a wet sludge and discharged into one of two boreholes feeding the underground receiving areas, located in a dedicated excavation near the bottom of the main decline.

The layout has been designed for the concrete mixer trucks to drive in from one side, collect the mixture, and exit the other side, providing a complete drive-through arrangement.

Refuge Stations

Refuge stations will be required to house underground mining personnel in a secure, hazard-free location during emergency conditions. Refuge stations will be provided underground in all working areas and travelways between working areas. No person will be farther than 750 m from the nearest refuge station in the event of an emergency. Two types will be used: permanent refuge stations in the underground workshops, and portable refuge stations for all underground areas away from the underground workshops.

Permanent refuge stations will be located near each underground maintenance workshop area. If the workshop refuge station is not serviced with suitable compressed air from surface, it will be equipped with self-contained breathing systems. In the event of an emergency, a notification system, with backup, will signal all personnel to stop work and proceed to the nearest refuge station. All refuge stations will be sized to meet the capacity requirements for the area.

Portable refuge stations will be completely self-sustaining and provide all required basic life support systems, creating a safe and secure ongoing environment for occupants. This includes oxygen supply, carbon dioxide (CO2), carbon monoxide (CO) scrubbing, cooling, and gas monitoring.

Portable refuge stations will be used to maintain compliance as the mining development faces advance.

Where DRC regulations do not address specific requirements for underground refuge stations, the South African Mine Health and Safety Act will apply. Refuge stations will comply with current regulations and legislation, including the Mine Health and Safety Act of 1996 (Act No. 29 of 1996).

The mine will issue an individual self-contained self-rescuer (SCSR) to each employee working underground and train them in their use.

Emergency Secondary Egress from Underground

General emergency escape plans will follow South African and US codes. Mobile personnel hoisting is considered a safe and efficient means of escape for circumstances that may be encountered underground. Proven rubber-tire hoisting systems using bullet-style conveyances, which can be lowered into ventilation raise boreholes, can effectively extract otherwise trapped personnel.

The following South African regulations are to be followed.

The two separate and independent shafts or outlets to surface required in terms of regulation 6.1.1:

a) shall not at any point be nearer to each other than 9.2 m,

b) shall be provided with proper arrangements, which will be kept constantly available for use, to enable persons to travel to and from the surface, and

c) shall be maintained in a safe condition and at a sufficient cross-sectional area throughout to allow for the free passage of persons.

Procedures for defining, evaluating, and reviewing the emergency escape system will be developed as part of the emergency escape strategy. Simulated emergency exercises (training) will be conducted at the mine at regular intervals.

Toilet System

Portable toilets will be used in strategic locations underground. A purpose-built toilet facility will be connected to each permanent refuge station at the underground workshops and sealable to the outside environment in the event of emergency. The toilets will be serviced by a mobile effluent removal cassette and transported to surface for discharge into the waste system by a mobile cassette carrier.

Power and Communication Systems

Electrical Substations and Power Distribution

The mine will be fed with two substations. Substation 1 will be installed during Phase 1 development and will supply developmental and permanent power. Substation 2 will supply the mine air cooling plants and backfill paste plants and will provide 11 kV power distribution.

Power will be distributed at 11 kV to the underground mine switchgear from two surface ring main feeder breakers, for redundancy. The underground switchgear will be contained in an E-room and will have separate feeder breakers feeding major mine areas for isolation purposes and to minimise large connected loads to each feeder. Each feeder will feed multiple mine power centers (MPCs), which will step down the voltage to 690 for centralised operational loads.

- Mine Medium-Voltage Distribution – 11 kV.
- Secondary Distribution – 690 V.
- Low-Voltage Distribution – 400/230 V.
- Frequency – 50 cycles per second.

Emergency Generator Plant

Emergency power will be supplied from the 20 MVA continuous- rated surface diesel generator plant on 11 kV to the following critical system loads under emergency conditions.

- Surface Ventilation Fans (6 MW).
- Underground Ventilation Fans (2.695 MW).
- Main Decline Dewatering Pumps (3.8 kW).
- Main Decline Rock Handling (6.25 MW).
- South Decline Dewatering Pumps of 3.8 MW.

Surface loads consist of surface production fans, backfill paste plants, booster stations, and the cooling plants for the underground mine.

Communications and Control/Automation Systems

The backbone for the communications system is based on a redundant fiber network. This system will be used to support all voice and data communication requirements for the Project. Radio communication for the mine will be provided over a leaky feeder system, which will be distributed throughout the entire mine for communication purposes, incorporating hand-held and fixed radios. This will be used to support the proximity detection system/vehicle detection system and ventilation-monitoring systems. The leaky feeder can also be used for central blasting.

The mining control room will be located on surface for monitoring/control of daily mining operations on surface and underground. The control system is based on a programmable logic controller (PLC) and supervisory control and data acquisition system. The main PLCs will be centrally located in the engineering rooms on surface with the exception of the main surface ventilation fans provided with a detailed PLC located within the substation at each vent shaft.

Underground Access Control

Upon entering the mine site through the surface access-controlled complex, mining personnel will proceed to the change house and lamp room where they will tag-in/tag-out. Miners' locations will be monitored in the control room by an electronic tracking system that is integrated into the cap lamps.

16.2.1.5 Equipment

All equipment will be sized for a 6 Mtpa case to support room-and-pillar and drift-and-fill mining methods. All ore material will ideally be conveyed out of the mine via a series of truck tips, ore passes, and conveyor belts. Waste material will be transported using mobile truck haulage equipment.

Criteria considered in equipment selection includes suitability, equipment standardization, 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 equipment requirements are split into two categories: mobile and fixed. The equipment requirements for each category are estimated at a prefeasibility level of accuracy and cover the major components for the operation.

Mobile Equipment

The mobile equipment will be diesel-powered, rubber tired. Typical development equipment such as jumbo drills will be used for the drilling and ground support. Explosives trucks will transport explosives and detonators to the headings. LHDs will load the blasted material and transport it to a remuck or the truck tips. LHDs will re-handle material transported to remucks into trucks where the material will be transported to truck tips or a designated area(s), depending on whether the rock is ore or waste.

In areas where the required development drift centerline height is 4.5 m high or less, low-profile equipment will be used (typically in the room-and-pillar and second lift areas of the drift-and-fill). Drifts that have 4.5 m or greater centerline heights (the first lift in the drift-and-fill areas and all primary development) will use standard equipment.

Initial and sustaining capital mobile equipment acquisition costs, rebuild costs, and replacement costs were calculated based on equipment life. Equipment life was calculated using operating hours as well as Vendor-provided actual operating hours for similar operations. Adjustments between engine (diesel) and electrical (e.g., hydraulics for drilling) hours were segregated.

The mobile equipment is listed in Table 16.17, which excludes the initial Contractor fleet.

Table 16.17 Mobile Equipment List

Description	Yearly Max. Req.	Purchase/Replace	Rebuild
Double-Boom Drill Rig – Standard Profile	22	66	52
Double-Boom Drill Rig – Low Profile	11	31	30
Cable Bolter	3	6	3
Haul Truck – 63 tonne	19	15	9
Haul Truck – 51 tonne	11	10	3
LHD – Standard Profile	19	55	20
LHD – Low Profile	12	21	15
Explosives Loading Truck	9	35	N/A
Explosives Transport Truck	5	19	N/A
Scaler	5	8	4
Skid Steer Cleanup LHD	10	37	N/A
Grader	3	8	N/A
UV	26	136	N/A
Concrete Mixer Truck	17	65	0
Large Personnel Transport	5	20	N/A
Telescopic Materials/Pallet Handler	6	41	N/A
LDV	34	215	N/A

Fixed Equipment

Table 16.18 lists the main fixed equipment that will support the mining operation at full production.

Table 16.18 Fixed Equipment

Services	Description	Services	Description
Materials Handling	Truck Tips	Safety and Miscellaneous	UG Safety Equipment
	Static Hydraulic Rock Breakers		Portable Refuge Stations
	Static Hydraulic Tramp Iron Magnets	Surface Facilities	Fuel and Lubrication Facility Equipment
	Apron and Vibrating Feeders		Concrete/Shotcrete Facility Equipment
	Conveyors		Temporary Emulsion Storage Facility Equipment
	Self-Cleaning Belt Magnets		Permanent Emulsion Storage Facility Equipment
Ventilation	Main Fans		Surface Heavy Vehicle Workshop Equipment
	Development/Production Fan		Surface Light Vehicle Workshop Equipment
	Mine-Air Cooling Facilities	Underground Facilities	UG Workshop – Central North Equipment
Mine Service Water	Centrifugal Pumps		UG Workshop – Central South Equipment
Mine Dewatering	Vertical Spindle (Sump) Pumps		UG Workshop – East Equipment
	Skid-Mounted Pumps and Tanks		UG Workshop – West Equipment
	Centrifugal Pumps		Satellite Shop Jib Cranes/Fire Doors
	Multistage Clearwater Pumps		Main Emulsion Storage Facility Equipment
Electrical and Communications	Main Substation		South Emulsion Storage Facility Equipment
	Motor Control Centers/Mine Power Centers		Concrete/Shotcrete Facility Equipment
	Leaky Feeder System		Fuel and Lubrication Facility Equipment

16.2.1.6 Personnel

Personnel requirements were developed to support development, construction, and operation requirements. Only personnel directly linked to the operation of the mine are included in this section. Personnel that share other Project activities (e.g., accounting, training, personnel management, environmental, permitting, housing, security, ambulance) are included in other sections of this report. Personnel requirements have not been determined for the EPCM Team.

Figure 16.27 illustrates the average annual personnel requirements for the mining over the LOM.

Figure 16.27 Contractor Versus Owner Personnel Summary

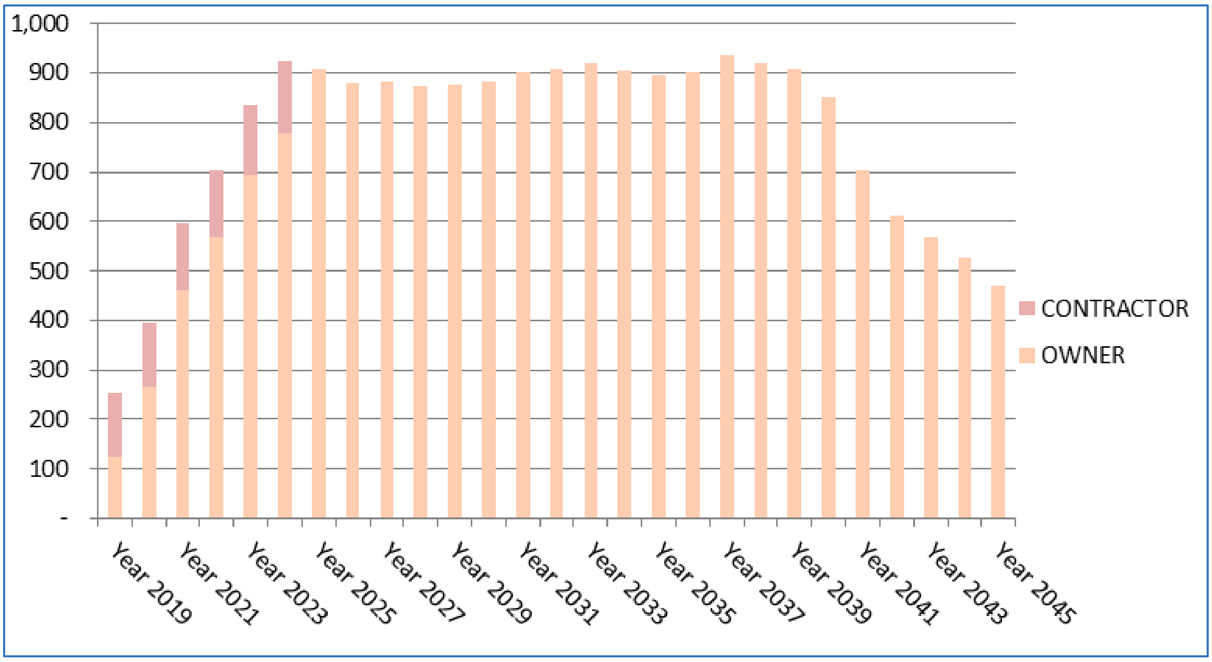


Figure by Stantec, 2019.

Direct and indirect labor requirements were determined based on the selected mining method, support systems, and general mine requirements during mine development, construction, and operations. Personnel requirements are based on a schedule of 12 h/shift and 2 shifts/d and 360 d/yr for both Contractor and Owner crews.

It is Stantec's understanding that workforce training is an extremely important requirement for the success of the Project; however, this basic training is courses covered in the overall human resources plan.

Personnel are presented as direct and indirect staffing for both Contractor and Owner teams. Staffing is further classified as expat hourly, expat staff, local hourly, and local staff.

The Owner's Project Team will oversee the work performed by the Contractor and coordinated by the EPCM Contractor. This includes labor, daily expenditures, and all equipment operating costs. Costs for the current and proposed Owner's Team, labor, and associated non-labor costs are included in the Study indirect costs. All production activities will be performed by Owner personnel.

During the preproduction period, Contractors will complete all construction and up to six crews for waste development activities. Contractor involvement in waste drift development will be complete in Q4 2024.

Additional allocations of personnel are included in the overall staffing, as follows.

- Payroll Personnel – Represents the daily personnel and the personnel that are on rotation but require inclusion in the overall personnel count. This includes direct supervisors as well as the underground hourly personnel.
- Vacation, Sickness, Absenteeism, and Training Allocation (VSAT) – Represents a “miner’s pool” that would be required on site to cover hourly labor during times when individuals are on vacation, sick, absent, or in training. This amounts to approximately 15% of the annual hourly personnel requirements.

16.2.2 Kamoa Underground Mining

The main mining methods include room-and-pillar for the mineralised zones above the 150 m depth and controlled convergence room-and-pillar below the 150 m depth.

The Kansoko Mine will be a mobile, trackless mining operation. Access to the mine is planned to be via a twin decline system from Kansoko Sud portal.

The Kamoa 2019 PFS production schedule has been developed based upon a 6 Mtpa production rate and high-grade scenario.

The overall mineral reserve could have been larger based on the size of the Kamoa deposit; however, only the targeted best 150.5 Mt in the Centrale and Sud regions were evaluated. From the targeted resource, a mine schedule that produced 125.2 Mt was produced.

16.2.2.1 Mining Methods

The Kamoa orebody geometry indicates different orebody thicknesses and slopes. The orebody dips between 0° and 35°, with an average dip of 17°. The thickness varies between 3.0 m and 6.0 m.

Access to the mine is via a twin declines system from the Kansoko Sud Portal, to support mining the Kamoa deposit. This has a total of 11,736 m of development. Besides primary development, the two mining methods for this orebody are room-and-pillar mining and controlled convergence room-and-pillar mining.

Room-and-Pillar

Room-and-pillar mining will be used in the mineralised zone between 60–150 m, to minimise the risk of surface subsidence. Continuing room-and-pillar mining below 150 m is required in selected areas for production ramp-up.

The production development of the room-and-pillar method will be in a grid-like fashion. The mining areas were divided to distinguish between the geotechnical needs for the room-and-pillar design above and below 150 m from surface elevation. The room development will run parallel to the strike of the panel for dips less than 20°, with belt drives running at an acute angle to the room drifts, to ensure the grade of the production drifts remains at or below minimum specifications. Where the dip is greater than 20°, the rooms will be developed slightly off the strike, to accommodate the acute angle between the room development and the belt drives.

Long-term stability is required for room-and-pillar to allow access for the miners while in production as the mining front begins at the access and progresses toward the ends of the panel. These room-and-pillar areas, designed to prevent subsidence, will remain accessible if maintained and ventilated.

The extraction ratios for room-and-pillar mining were based on the dip and the height of the panels and Cuprum's resulting pillar design.

At a depth of 100 m, pillars are required to be 9 m x 9 m with 10 m-wide rooms. A mining height of 6 m has been assumed. Eight rooms will be required to meet the maximum panel span of 152 m, which will be bounded by 20 m regional pillars. A maximum strike length of 504 m has been allowed. A row of regional pillars 15 m wide will then be required before the next panel is started.

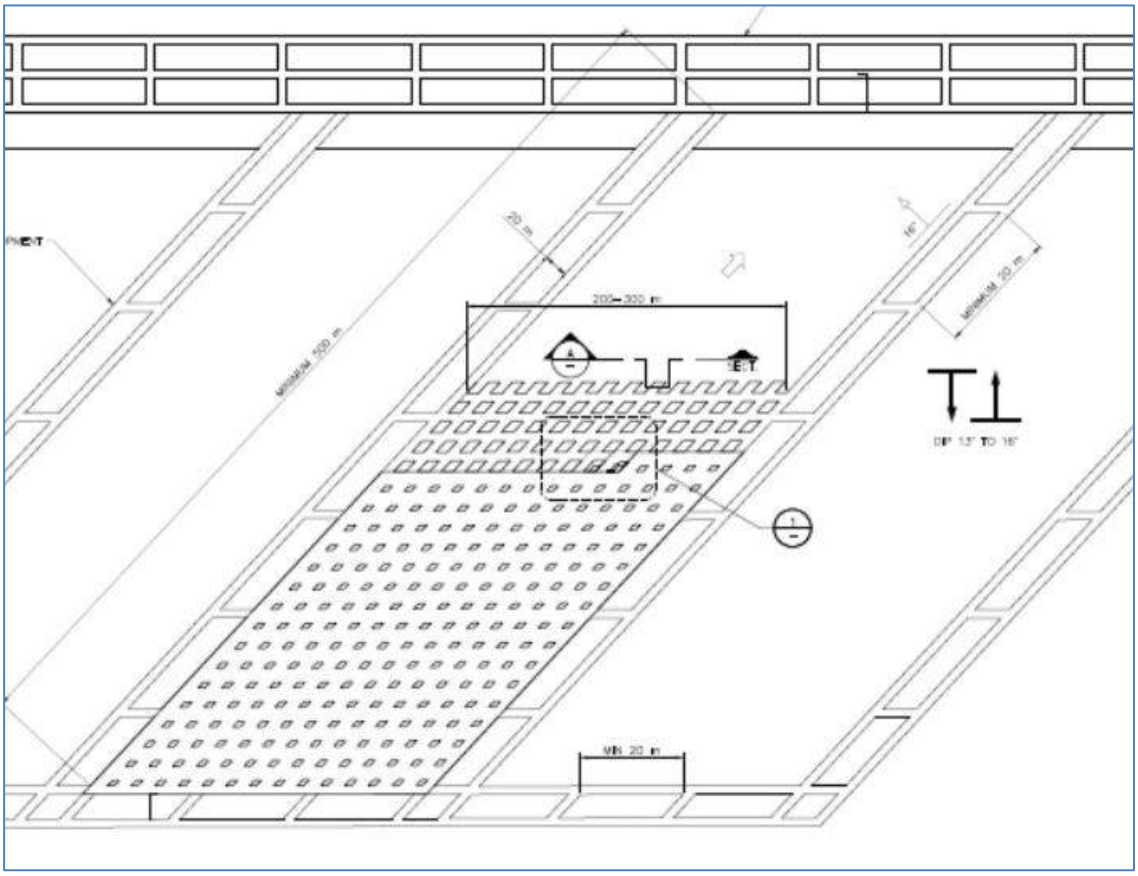
The calculated extraction ratios for areas mined using the room-and-pillar method vary with depth and range from 53% to 70%.

Controlled Convergence Room-and-Pillar

Controlled convergence room-and-pillar mining will be used in the mineralised zone below 150 m. An initial panel will be taken as a trial panel to confirm the method viability. Once 70% complete, the additional panels will start production mining.

The development of the panel requires secondary drifts to be excavated on the perimeter, to allow access of equipment to the production headings. This secondary development will consist of two headings connected by cross-cuts. If the panel is being mined from the extents toward the access, then the secondary development will be driven completely around the perimeter of the panels. The panel dimensions are generally 300 m wide and a minimum of 500 m long, where possible. In the case where the mining front is progressing away from the access, the perimeter development will only be designed along the sides of a panel. If truck haulage is required for the panel, the secondary development will be large enough to allow trucks to be driven into the panels. Figure 16.28 is a typical controlled convergence room-and-pillar panel, with a mining direction advancing toward the access.

Figure 16.28 Typical Controlled Convergence Room-and-Pillar Mining Panel



Upon completion of the required secondary development, production development will begin by establishing room drifts and their associated belt drifts. Similar to room-and-pillar mining, the angle between these drifts are determined by the dip and thickness of the orebody. The angles will accommodate the maximum gradient permissible in this design, which is 12° or less.

During the retreat of a final panel where a panel adjacent has been mined out, the belt drives will carry into the secondary development nearest to that panel as part of the mining front.

The room and belt drives will form technological pillars. These pillars are designed to compress the load of the working back. As the mining working area increases, the pillars take more stress and cause the convergence. It has been determined that the maximum mining area to activate is no more than three-belts distance from the working face to the pillar scraping.

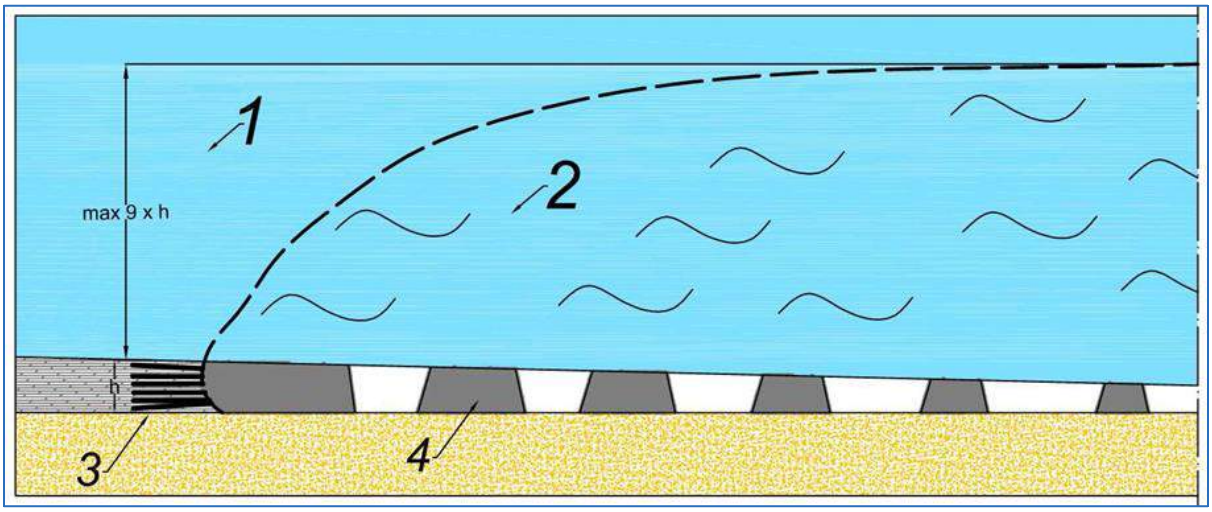
These technological pillars will be reduced through scraping or drill and blast to a remnant pillar size. The remnant size is the minimum size pillar that is allowed to be scraped while still ensuring the overall front is supported and continues to converge. Once remnant size pillars are established, personnel and equipment will be prevented from working in that area.

Controlled convergence room-and-pillar mining is currently utilised by KGHM at their mines in Poland. It is based on the strength and strain parameters of the rock that make up the mining panel supporting pillar or technological pillars and includes the following parameters:

- Ore zone depths below 150 m.
- Strength of the immediate roof (i.e. roof bolting and handling of the rock burst threat).
- Strength and strain parameters of the rocks within the roof of the extraction panel (i.e. the slow bending above the extraction space and in the workings).
- Technological pillars (pillars between rooms) designed to work in the post-destruction strength state to maximise ore extraction.
- Cuprum has developed the controlled convergence room-and-pillar methodology at its mines in Poland and are the technical contributors to its adaptation for the Project.

Extraction mining with roof deflection and pillar strength in the post-destructive state is based on a modified Labasse hypothesis (1949) (see Figure 16.29). The relationship between the pillar height-to-width ratio should be within the range of 0.5–0.8. This ensures the progressive transition of the technological pillars into the post-destructive strength state, enabling a smooth roof-bending strata (destressed and delaminated rock mass) above the workings. The extraction ratios for controlled convergence room-and-pillar mining were based on the dip and the height of the panels and Cuprum's resulting pillar design.

Figure 16.29 Controlled Convergence Room-and-Pillar Rock Mass Impact



1 – rock mass prior to extraction; 2 – destressed and delaminated rock mass; 3 - blasting holes; 4 - primary pillars. Figure by KGHM Cuprum, 2017.

16.2.2.2 Mining Dilution and Recovery Factors

To obtain dilution grades, three dilution shells were constructed around each production panel shape to report the grade and density outside of the targeted resource. The three shells comprise a 1.0 m hangingwall (HW1) dilution shell on top of the production panel shape and 2 x 1.0 m footwall (FW1, FW2) dilution shells on the bottom. The block model interrogated the dilution shells by block centre to provide the dilution grades and densities for each shell.

The grades were then applied to the calculated tonnage of dilution for each production shape. Primary development includes the two service drifts with a cross-sectional dimension of 5.5 m wide x 6.0 m high and a conveyor drift with a cross-sectional dimensions of 7.0 m wide x 6.0 m high. The development headings were assumed to have a flat back with arched corners and will not be affected by the height or spatial location of the grade shell, which would result in a fluctuating grade and planned dilution percentage in each segment for the length of the drift. Back and wall dilution is assumed to have an average overbreak of 0.1 m.

Room-and-pillar production only includes production from the room-and-belt drifts. There is no pillar extraction for this method. For hangingwall dilution, overbreak is assumed to average 0.15 m. No dilution from the walls was considered, since the pillar width must be maintained. Controlled blasting practices will be required to ensure that the walls are broken to design width. The footwall dilution is a planned dilution and is based on the dip and thickness of the production panel shape. For thicknesses where the short side wall is less than 2.5 m high, the angle of the back and the floor are adjusted. The result is a slightly increased footwall dilution and slightly reduced hangingwall dilution. This occurs for all 3.0 m high thicknesses with a dip greater than 12° and all 3.5 m high thicknesses with a dip greater than 16°.

Controlled convergence room-and-pillar production includes the development associated with panel perimeter drifts (secondary development) and production from the room-and-belt drifts plus pillar extraction. For hangingwall dilution, overbreak is assumed to average 0.15 m. No dilution from the walls was considered since the overbreak is mainly from pillars, which will be extracted later in the production cycle. The hangingwall overbreak dilution is expected to project across the back of the pillar as it is extracted. The footwall dilution is a planned dilution and is based on the dip and thickness of the production panel shape. Ore from the secondary panel drifts was captured in the extraction ratio tonnage calculations and therefore was not counted in the development tonnage.

In secondary development where the back height is restricting truck access, a decision was made to increase the back height, which increases the amount of dilution associated with the hangingwall. The truck height for this evaluation was 3.86 m from the sill to the top of the fully loaded truck. The restriction occurs for drift heights of 3.5 m and lower.

Mining Recovery

The mining recovery includes allowances for equipment limitations, heading shapes, heading strike and dip angles, ore re-handling, and operator skill. For primary development, the recovery is 98%. Lost tonnage is a result of losses due to the corners of the drift and muck that settle into the irregularities in the floor. Stantec estimated 0.1 m of rock material will be lost on the floor.

Room-and-pillar mining is development intensive and will have recoveries similar to primary development. Some material will be left along the corners of the walls but will be recovered during the pillar extraction phase, so a recovery of 98% is expected.

Controlled convergence room-and-pillar mining is similar to room-and-pillar mining as it is development intensive and will also have recoveries similar to primary development. Some material will be left along the corners of the walls but will be recovered during the pillar extraction phase, so a recovery of 98% is expected. Due to the space of the working area around the pillar and the larger muck size created from the scraping process of extraction, the initial recoveries will remain high. Ore extraction losses will occur when pillars cannot be completely recovered due to deterioration of ground conditions and steep dips. Based on these factors and experience, a 95% mining recovery was applied to pillar extraction tonnages.

16.2.2.3 Mining Access Design

Mine access is required to ensure safe and reliable transport of mining personnel and equipment, for production, for intake and exhaust ventilation-ways, and to facilitate the reticulation of all services to and from the mine workings.

Key access design objectives were to:

- Access the workings in a way that minimises capital development.
- Facilitate an aggressive production build-up, targeting the high-grade areas as quickly as possible.

Access into the mine will be via a set of twin declines from the portal down to the Kansoko Sud/Centrale breakaway. One decline will house the main conveyor and the other will be used as the service decline. The declines from the surface will be inclined at -8.5°, considered the optimal inclination for mechanised equipment. The conveyor decline will extend beyond the Kansoko Sud/Centrale breakaway to the storage silo system. The conveyor decline inclination will increase to -12° to allow construction of the storage silos below the ore horizon. From the top of the storage silo system, the Kansoko Sud conveyor decline will be developed to the south to the most southerly Kansoko Sud mining block.

The service decline will terminate at the Kansoko Sud/Centrale breakaway, and a set of triple declines will be developed down the Kansoko Sud/Centrale access to the breakaway of the Kansoko Sud roadway. Triple declines will then be developed into the Centrale North and South mining areas, and a twin roadway system will develop into the Kansoko Sud mining area.

Development dimensions will be 5.5 m wide x 6.0 m high for the service drift and 7.0 m (W) x 6.0 m (H) for the conveyor drift, based on the conveyor design, ventilation intake requirements, and sizes of equipment.

The portal is positioned to facilitate quick access to the shallower parts of the orebody and to the higher-grade areas of the Kansoko Sud mining area. It also allows early development towards the high-grade areas of the Centrale mining area. Figure 16.30 shows the portal, declines, and underground infrastructure.

Figure 16.30 Kansoko Mine Underground Access Infrastructure

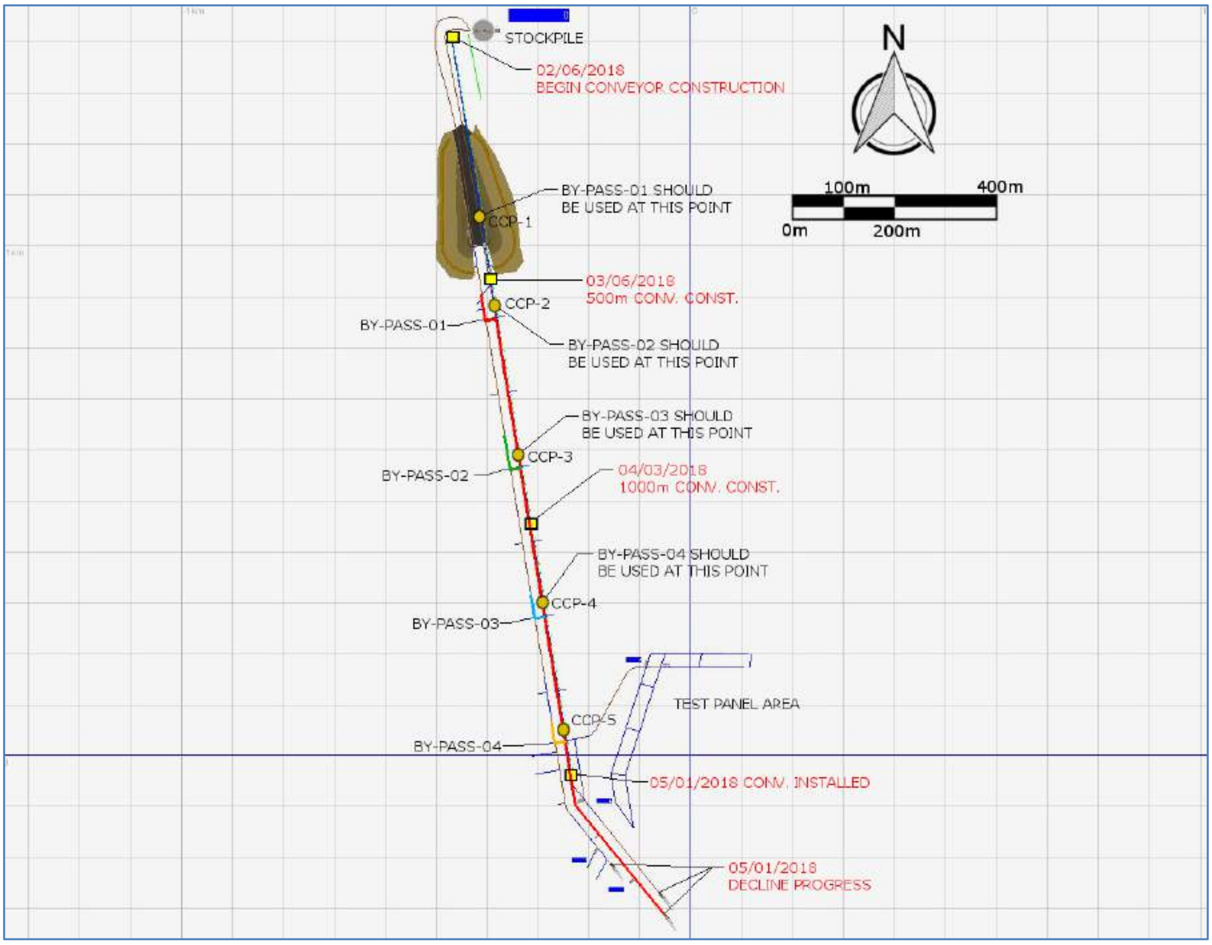


Figure by Kamoa Copper SA, 2015.

16.2.2.4 Mining Schedule

Development and Construction Schedule

The development schedule focuses on establishing necessary mine services and support infrastructure to set up the initial production mining areas and to ramp up to 6 Mtpa ore production and associated development waste. The full production schedule will be based on a 360-day calendar that will be sustained for 17 years with a 26-year LOM.

Mine development will occur in the following three main phases:

- Phase 1: Development of the Declines to the Main Ore Bins.
- Phase 2: Controlled Convergence Room-and-Pillar Initial Panel and Room-and-Pillar Mining.
- Phase 3: Development of Centrale and Sud.

Table 16.19 summarises the LOM development and production.

Table 16.19 LOM Development and Production Summary

Waste Development	
Lateral Development (m)	31,390
Lateral Development Tonnes (t)	3,297,172
Mass Excavation Lateral Equivalent (m)	2,517
Mass Excavation Tonnes (t)	224,276
Vertical Development(m)	5,182
Vertical Development Tonnes (t)	296,952
Total Waste Development	
Metres (m)	39,089
Tonnes (t)	3,818,399
Production by Mining Method	
Ore Development Metres (m)	114,205
Ore Development Tonnes (t)	10,665,176
Room-and-Pillar Production (m)	36,743
Room-and-Pillar Production (t)	3,396,830
Controlled Convergence Room-and-Pillar Production (m)	813,559
Controlled Convergence Room-and-Pillar Production (t)	111,120,408
Total Ore Production	
Total Ore Development (m)	114,205
Total Production (m)	850,302
Total Tonnes (t)	125,182,414
Diluted Grade	
NSR ($/t)	$168.06
TCu (%)	3.81
AsCu (%)	0.32
S (%)	2.49
As (%)	0.00
Fe (%)	6.14
Density (t/m^3)	2.93

The following criteria were applied over the mine life for scheduling purposes:

- Proximity to the Main Accesses and Early Development.
- High-grade and Thickness.
- Ventilation Constraints.
- Mining Direction.
- 300 m Gap Distance between Two Adjacent Panels Fronts.
- Application of a Declining Cut-off Grade.

Using the above strategy, appropriate panels were targeted and scheduled to achieve the highest possible grade profile during ramp-up and full production.

Mine Development Plan and Scheduling

For primary development, the rates in Table 16.20 were calculated using first principles. Cycle inputs were obtained from various sources (e.g. original equipment manufacturer [OEM], external consultants, specialists) and compared with Stantec inputs. The cycles were updated accordingly following team discussions.

Table 16.20 Primary Development Rates

Description	Single-Heading Performance (m/d)	Double-Heading Performance (m/d)	Multi-Heading Performance (m/d)	Single-Heading Performance (t/d)	Double-Heading Performance (t/d)	Multi-Heading Performance (t/d)
5.5 W x 6.0 H – Semi-Arch (Service Drifts)	3.96	5.35	5.94	395	533	592
5.5 W x 6.0 H – Flat (Cross-cut Drift)	3.93	5.31	5.89	389	525	584
7.0 W x 6.0 H – Semi-Arch (Conveyor Drifts)	3.49	N/A	N/A	427	N/A	N/A

The zero-based rate calculations for secondary and production drift development in controlled convergence room-and-pillar mining were developed based on the drift cross-sections that have a plan view width of 7.0 m, with wall slopes 10° from vertical. Room-and-pillar mining also has a width of 7.0 m but has vertical walls. Secondary drifting is not required for room-and-pillar mining. Secondary and production drifts generally follow the ore, with minor additional cross-section enlargement into the waste of the back and floor for thin, steeply dipping areas. For drift heights less than 4.0 m at the centre of the drift, the back height will be increased to accommodate a haul truck where required.

The secondary and production drift cross-sections (with controlled convergence room-and-pillar) provided by Cuprum were supplemented with cross-sections for 4.0 m and 5.0 m high drifts, so that consistent half-metre height increments could be used. Changes in the cross-sectional area and perimeter were analysed with changes in dip. It was determined that drifts with the same drift height and dips <20° could be combined and represented by the average area and average perimeter within a few percentage points variance, which is within the required accuracy of the study. Only 4.0 m and 6.0 m high drift sizes in the ≥30°–35° dip categories were in the mine design, so only these drift size productivities were calculated in this dip range.

The room-and-pillar production drifts for room-and-pillar production with non-convergence have inclined backs parallel with the dip and flat floors like the convergence production panels, except that the ribs are vertical instead of canted.

Preproduction Development Schedule

The initial development in the ramp-up periods will require a significant amount of the overall waste development that will be mined for the Kansoko Centrale and Sud mining areas. The waste development consists of the main infrastructure such as conveyor excavation, main shops and infrastructure, and dewatering settlers. Most ore development in this period consists of the service and conveyor declines, room-and-pillar mining, and the secondary development in preparation of panel production. Figure 16.31 illustrates the preproduction development schedule.

Figure 16.31 Preproduction Development Schedule

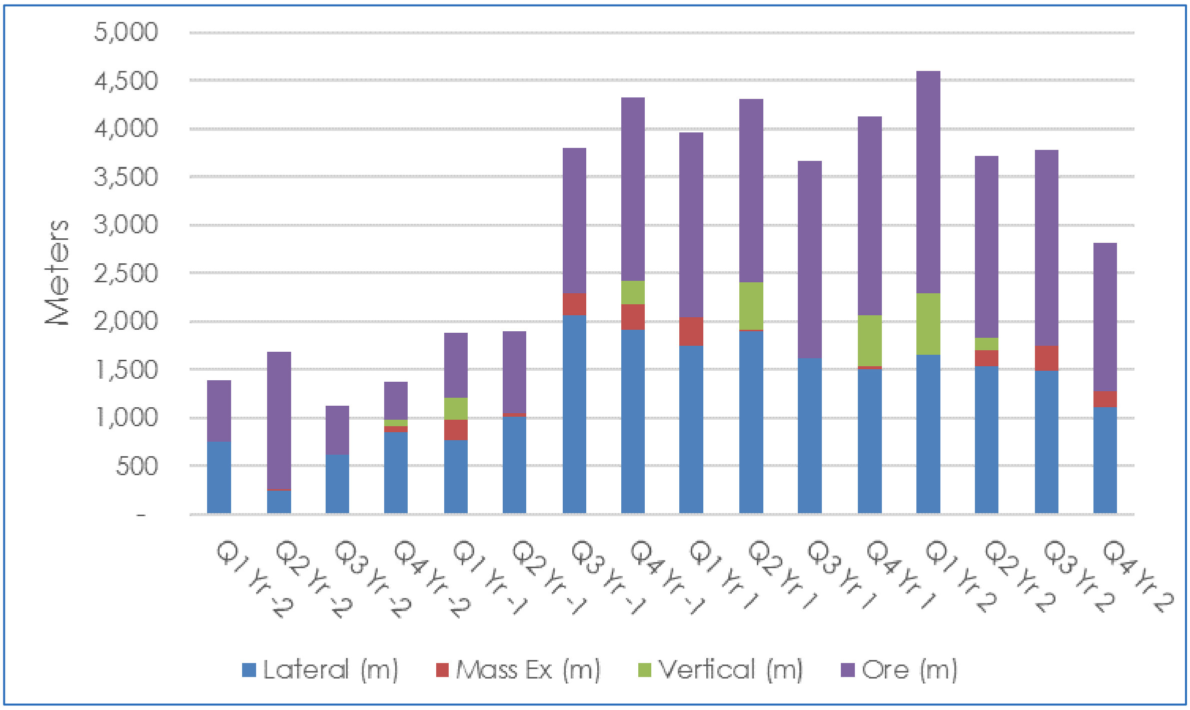


Figure by Stantec, 2017.

Life-of-Mine Development Schedule

The development schedule beyond the initial preproduction period targets the areas required to bring online the production panels that support the LOM plan. This would include excavating the primary and conveyor drifts ahead of production panels to access necessary ventilation raises. Figure 16.32 illustrates the LOM development schedule.

Figure 16.32 Life-of-Mine Development Schedule

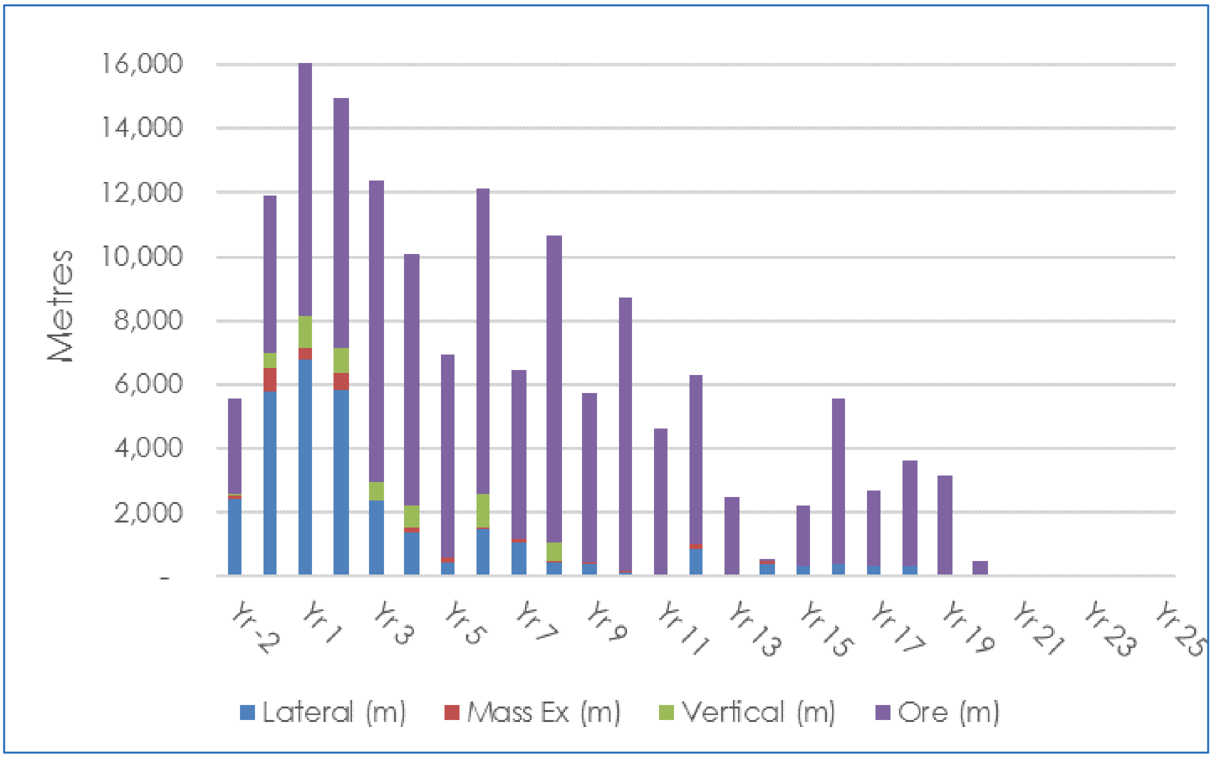


Figure by Stantec, 2017.

Mine Production Plan and Scheduling

The development schedule focuses on the establishment of necessary mine services and support infrastructure to set up the initial production mining areas and ramp-up to 6 Mtpa ore production and associated development waste. The full production schedule will be based on a 360-day calendar that will be sustained for 17 years with a 26-year LOM.

The following criteria were established for the targeted resource, to support the overall tonnage requirements from the Kamoa deposit. Table 16.21 details the targeted annual tonnages for the overall production requirements to meet the 6 Mtpa production rate.

Table 16.21 Production Schedule Criteria

Criteria	Details	
Initial and Ramp-Up	10.9 Mt	
Full Production	6 Mtpa	
Extraction / Recovery	75%	
Production Schedule	**Years**	**Tonnes**
Initial Production Mining (Year -2)	1	200,000
Ramp-Up (Year -1)	1	1,200,000
Ramp-Up (Year 1)	1	2,000,000
Ramp-Up (Year 2)	1	3,000,000
Ramp-Up (Year 3)	1	4,500,000
Full Production	17	102,000,000
Production (prior to ramp down)	22	112,900,000

Full production of 6 Mtpa is 17 years and tapers off as the reserve is depleted. The panels were scheduled so that a higher NSR value is achieved earlier in the project. Figure 16.33 illustrates LOM schedule and grades. The mine production schedule is detailed in Table 16.22.

Figure 16.33 Life-of-Mine Schedule and NSR

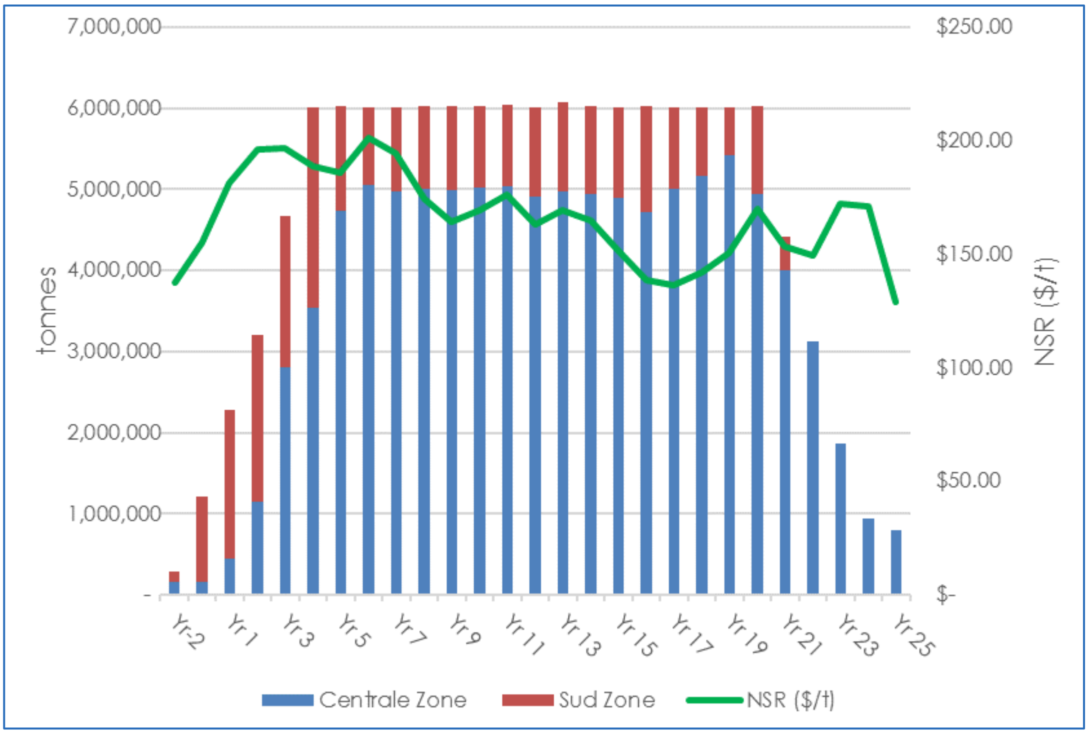


Figure by Stantec, 2017.

Table 16.22 Mine Production Schedule

Description	Unit	Total	Project Time (Years)													
			-2	-1	1	2	3	4	5	6	7	8	9	10	11	12
Room-and-Pillar Ore Mined	(kt)	3,397	–	–	347	656	1,033	446	640	275	–	–	–	–	–	–
	(% Cu)	5.29	–	–	5.49	6.12	4.82	6.28	4.92	4.09	–	–	–	–	–	–
Controlled Convergence Room-and-Pillar Ore Mined	(kt)	111,120	–	–	396	897	1,400	3,302	4,524	5,163	5,086	5,505	5,145	5,549	5,260	5,605
	(% Cu)	3.81	–	0.00	2.85	3.57	4.50	4.45	4.20	4.31	4.65	4.50	4.06	3.77	3.98	4.02
Ore Development Ore Mined	(kt)	10,665	–	289	464	729	772	919	850	589	923	510	882	478	767	445
	(% Cu)	3.34	–	3.15	2.63	2.83	3.68	3.44	4.06	3.37	4.00	3.20	3.42	3.55	2.97	3.65
Total Ore Mined	(kt)	125,182	–	289	1,206	2,282	3,205	4,667	6,014	6,028	6,010	6,015	6,027	6,027	6,027	6,050
	(% Cu)	3.81	–	3.15	3.52	4.07	4.40	4.42	4.26	4.21	4.55	4.39	3.97	3.75	3.85	3.99

Description	Unit	Total	Project Time (Years)													
			13	14	15	16	17	18	19	20	21	22	23	24	25	26
Room-and-Pillar Ore Mined	(kt)	3,396	–	–	–	–	–	–	–	–	–	–	–	–	–	–
	(% Cu)	5.29	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Controlled Convergence Room-and-Pillar Ore Mined	(kt)	111,120	5,560	5,862	6,021	5,818	5,613	5,813	5,731	5,751	5,974	4,417	3,126	1,858	943	802
	(% Cu)	3.81	3.78	3.87	3.76	3.46	3.20	3.12	3.26	3.47	3.86	3.49	3.42	3.89	3.83	2.93
Ore Development Ore Mined	(kt)	10,665	448	215	1	187	406	198	278	267	48	–	–	–	–	–
	(% Cu)	3.34	2.80	3.29	1.93	3.44	2.79	3.01	2.82	2.83	3.54	–	–	–	–	–
Total Ore Mined	**(kt)**	125,182	6,007	6,077	6,022	6,005	6,019	6,012	6,009	6,018	6,022	4,417	3,126	1,858	943	802
	(% Cu)	3.81	3.71	3.85	3.76	3.46	3.17	3.12	3.24	3.44	3.86	3.49	3.42	3.89	3.83	2.93

Totals may not add up due to rounding.

16.2.2.5 Underground Infrastructure

Ventilation

Diesel particle matter is the main driver for establishing airflow requirements for the underground openings as the baseline. These requirements will be later adjusted for the required cooling and refrigeration, increasing the cooling capacity of the ventilation system. To do this, initially the mine development and production schedule, in conjunction with underground equipment, will be considered to determine the required air quantity and primary flow distributions. Heat load calculations along with computer simulations with the variables obtained from the mine design will determine mine air cooling and refrigeration requirements. The strategy will be to ventilate the mining sections with flow-through ventilation and avoidance of recirculation/reuse of air. Main service and conveyor declines will provide fresh air, while a ventilation raise near the bottom of the main declines (Vent Raise No. 1) will be used as an exhaust column. Fresh air from the declines will split into two major development fronts—one supporting Centrale and the other Sud. Bulkheads, ventilation curtains, seals, pillar sections, and booster fans will be used to control the air distribution within the panels.

The ventilation system is designed to provide localised fresh air intake for the major mining areas—Sud, Centrale North, and Centrale South—with dedicated exhaust assigned for each of the mining areas. The fans are located at the exhaust shafts on surface where possible, to reduce heat gain in the fresh air supply.

During main access decline development, the main service and main conveyor declines will be driven blind to every other cross-cut, where the cross-cut will be used to establish a loop for the next blind segment. When the loop is created, the service decline will be used as the return airways and the conveyor decline will be the intake air. During the development of the blind headings, an exhaust overlap system will be used for maximum performance and safety.

The initial exhaust ventilation raise (Vent Raise No. 1) located at the bottom of the main declines will provide required ventilation to Centrale and Sud primary development headings. The main twin declines will provide fresh air, and the return flow will be exhausted through the ventilation raise.

As the mine life progresses, additional intake and exhaust ventilation raises will be developed to meet demand. All major ventilation fans, except those venting Vent Raise No. 1, will be installed at surface.

The primary development's "triple-line" to the mining areas will deliver fresh air as a flow-through system. Two primary service drifts and one primary conveyor drift will be driven blind to every other cross-cut, where the cross-cut will be used to establish a loop for the next blind segment. Each of the mining areas is designed to have a dedicated fresh intake shaft servicing Sud, Centrale North, and Centrale South.

The secondary drifts define the panel and will be the primary ventilation route for the panel. These will be developed following the contour elevation of the ore, orientated close to the strike. These will be twin headings with cross-cuts between the drifts. During the development of the secondaries, one heading will be the intake and the other will function as an exhaust.

Typical mining direction within the Controlled Convergence Room-and-Pillar panels will begin at the extremities. Rooms and belts will be mined adjacent to the secondary at the extremities of the panel to establish flow-through ventilation. The typical ventilation circuit will flow through the secondary drifts into the active mining area once the connection is established. Fresh air will flush over the mining face due to the negative pressure from the exhaust side connected to an exhaust raise. Air distribution within the rooms will be controlled with ventilation seals, curtains, pillar sections, and jet fans including booster fans.

Cross-cuts not requiring future access may be sealed with shotcrete walls. Primary fresh and exhaust airways are considered long-term development and will require corresponding long-term ground support. A combination of regulators and air doors, along with auxiliary booster fans, will direct airflow to the active mining areas. The possibility of using ventilation on demand should also be explored.

The airflow required takes into consideration the utilisation factor of the mobile equipment and is rated at 0.063 m^3/s per brake horsepower, with utilisation factors applied. The equipment shows the crew requirement for development, production (room-and-pillar and controlled convergence room-and-pillar), and haulage of rock (ore and/or waste). The leakage throughout the mine was taken to be 15%, requiring a total flow of approximately 1,424 m^3/s at full production.

This total flow requirement assumes that in the full production scenario, eight panels are active, and four development headings are being driven, with five trucks to move material outside of panels.

Mine Air Cooling Facilities

Refrigeration will be required to provide sufficient cold air and to ensure that the development and panel reject temperatures remain within design parameters (i.e. average development and panel reject wet bulb temperatures of 28.5°C). A first-order comparison of alternative refrigeration systems, notably underground refrigeration installation and surface ice makers, showed that the surface refrigeration using chilled water was the most economical. Ice systems are approximately 1.2 times the cost of normal refrigeration systems (underground melting dams, shaft pipes, pumping, remote heat exchangers, etc.), and underground refrigeration systems cannot be justified from an efficiency and operational perspective.

The BAC system will be installed on surface. The BAC system will be a horizontal-type, counter-flow, three-stage heat exchanger. Fresh air will be forced into the BAC chamber by means of three 200 m^3/second force fans positioned in parallel on the intake side of the BAC (600 m^3/second total). The intake mean summer wet bulb temperature will be 20°C, and the design outlet air temperature will be 10°C saturated. The BAC will be a direct contact-type system that provides the maximum heat transfer efficiency required. Chilled water will be sprayed into the moving air within the BAC chamber by means of spray nozzles separated in equal spacing along the chilled water pipes. The water droplets will fall to the bottom of the chamber and be reticulated back to the refrigeration plant evaporator plate heat exchanger system where the system is repeated.

To avoid overland piping and interference with community infrastructure, Kamoa's preference is for independent surface cooling installations. The proposal therefore is to locate discrete refrigeration plant rooms and heat rejection facilities at Vent Raise Nos. 7 and 4. The Vent Raise No. 7 plant will be sized for a nominal BAC duty of 10.0 MW; the plant at Vent Raise No. 4 will be sized for a nominal BAC duty of 4.0 MW.

Ore and Waste Handling Systems

Underground ore and waste handling will be designed for rubber-tyred and conveyor belt transportation of broken ore and associated waste, 360 d/yr. LHDs or haul trucks will transport the rock from the working headings. Ore will be moved to the surface via conveyor. Waste rock will be moved to surface using truck haulage, conveyor, or will be cast underground into the mined-out room-and-pillar areas.

The mine will be a mobile, trackless operation designed with a 6 Mtpa capacity for ore and waste handling. Bulk transport of ore and waste from the mining areas to the primary underground storage silos will be via a network of conveyor belts. LHDs and haul trucks will be used to transport ore from the mining panels, through tips, onto conveyors in the Centrale North, Centrale South, and Sud mining areas, respectively. Waste will be trucked out the mine directly to the waste dump on surface or into mined out areas underground.

The conveyors from Centrale North and Centrale South will converge onto a single conveyor that feeds an underground silo. Similarly, the conveyor belts from the Sud area will feed a second underground silo. Ore will be fed through the silo onto a transfer belt, which will feed the main decline belts transporting the ore to surface. The conveyor head pulley of the final belt in the decline system will discharge into a splitter discharge chute for transfer of materials onto the process plant feed conveyor or onto a shuttle conveyor, distributing the rock onto one of two ground stockpiles—a waste stockpile and an ore stockpile.

There will be two ore passes located in the Sud zone where the conveyor drift is located beneath the orebody. They will be equipped with fixed hydraulic rock breakers and sizing grizzlies (panel grizzlies) nested at the top of the ore pass. Mechanical feeders will transfer material at a controlled flow rate onto the conveyor.

Bins and Transfer Points

Production from the Sud and Centrale mining areas will feed into two vertical storage silos positioned at the bottom of the main decline conveyor (No. 1). The silos will control and regulate the feed onto the main decline conveyor belts and will also provide storage capacity to allow for maintenance of the belt and/or failure of one of the conveyor belts.

Each silo will have a dedicated feed from the Sud and Centrale mining areas, respectively. During the initial development phase of works, a haul truck tip arrangement will be available at the top of the silos, complete with a grizzly and rock breaker. This arrangement facilitates loading of the initial development rock into the silos, until the conveyor belts are installed and commissioned. Both silos are 13 m high x 6 m in diameter, with a live capacity of 750 tonnes. The silos will be lined so that self-mining after prolonged use does not occur. The silos will be lined with 40 MPa concrete and then painted with a sodium silicate cover, which will give the lining a final finished hardness of approximately 65 MPa.

There are grizzlies situated at the top of each of the ore passes, which will be identical throughout the ore handling system. Tramp iron will be collected by hand.

The primary bulk handling system comprises a network of conveyor belts in the following locations:

- Main Decline.
- Sud Conveyor Drift.
- Centrale North Conveyor Drift.
- Centrale South Conveyor Drift.
- Centrale Conveyor Drift.

Figure 16.34 indicates an overall layout of the conveyor belt network.

Figure 16.34 Conveyor Network System

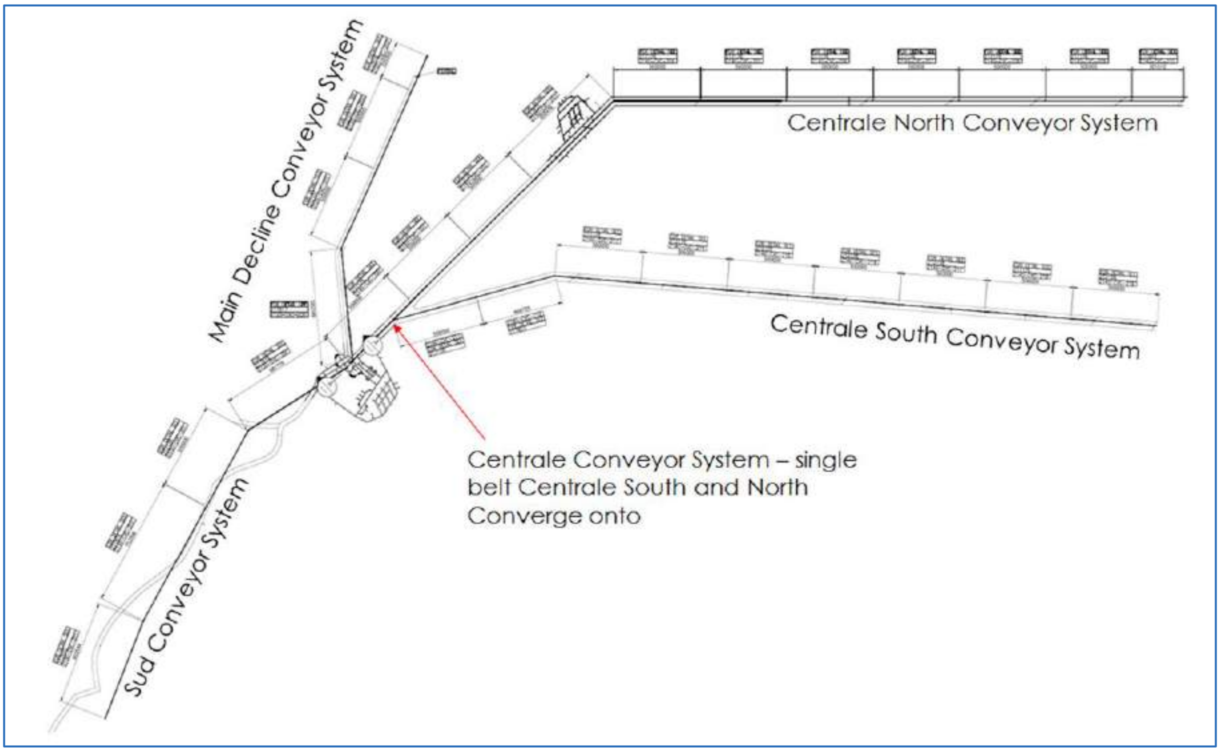


Figure by Stantec, 2017.

All the decline conveyors are designed to convey the total mine ore production of 6 Mtpa. Thus, all conveyors will have a capacity of 1,875 t/h and a belt speed not exceeding 2.5 m/s.

For the Sud conveyor system, ore will be transported from the face to two ore passes by dump trucks and LHDs. The ore passes will feed down to the Sud decline conveyor. Ore is also loaded with an LHD through a grizzly onto one of three 15 m long x 1,500 mm wide Class 2000 sacrificial belts. The belts support the two on-reef conveyor loading points within the Sud area, which feed the main Sud conveyor belt. The belt width standardises belting, components and helps prevent spillage during loading.

The Sud conveyor system comprises four conveyor belts in series. All Sud conveyors are designed to convey ore at a rate of 6,000 t/d (mined out of two panels). All Sud conveyors will have a capacity of 488 t/h and a belt speed not exceeding 1.6 m/s. Conveyor Nos. 1, 2, 3, and 4, operating in a series arrangement, will transfer material from the ore passes and on-reef conveyor loading points into the Sud underground silo.

For the Central North and South conveyor systems, ore will be transported from the face to the ore passes by haul trucks and LHDs. The ore passes will feed down to either the Centrale North or Centrale South decline conveyors.

The Centrale North conveyor system comprises 10 conveyor belts in series, and the Centrale South conveyor system comprises nine conveyor belts in series. All conveyors are designed to convey 12,000 t/d (mined out of four panels) ore production in Centrale North and Centrale South, respectively, with 900 t/h capacities and belt speeds not exceeding 1.6 m/s. The ore and waste from Centrale North and Centrale South will then converge onto an 1,800 t/h capacity conveyor belt that feeds the underground silo.

Each of the ore passes will have a bulkhead containing a feed arrangement that feeds the respective conveyor belt. Haul trucks or LHDs will discharge their loads through the grizzly and into the ore pass. A vibrating feeder will feed the rock onto the conveyor belt.

Workshops

Major mobile equipment will remain underground for the duration of the machine's life cycle and will be serviced/maintained in applicable underground workshops. Machines will only come out of the mine for a complete OEM refurbishment, or to be scrapped and replaced.

The final mine layout comprises two main workshops (main and Centrale) and several satellite workshops. The main underground workshop will be located near the intersection of the main decline and the primary accesses to the Sud and Centrale deposits. This underground workshop is central to both production mining areas. The Centrale workshop will be located near the bend of the northern decline at Centrale to reduce travelling distances from working places at Centrale.

As the mining progresses and travel distances increase, satellite workshops will be established near to production areas and furnished with the appropriate service equipment.

Production fleet vehicles operating mainly at the production face (drill rigs, bolters, and LHDs) will be serviced and maintained (minor repairs) at satellite workshops. All vehicles will revert to the main workshop for major services/repairs. Trucks hauling waste to surface as well as UVs will be maintained in the surface workshop. The surface workshop will also be equipped for rebuilds.

The bulk of tyres will be stored on surface. Minimum quantities of tyres will be stored underground in the allocated bay. Tyre repairs and fitting to the required rims will be undertaken on surface and transported to and from the workshops daily as required. The tyre bay is equipped with a 5-tonne overhead crane.

The multi-purpose vehicle (MPV) is equipped with a tyre handler to assist with changing wheels at the workplace or point of breakdown. The MPV will collect the tyre from the tyre bay and return the used/damaged tyre to the bay. The bay is equipped with racks for various rimmed tyres.

Fuel and Lubricant Distribution

The mine will operate as a trackless mining operation. The supply of fuel and lubricants is necessary for the operation of diesel-powered mobile and underground fixed mining equipment.

A diesel and lubrication storage and distribution facility with refueling pumps will be constructed on surface, near the portal. These will be used initially until the facilities at the main workshop have been completed. The initial surface installation will be used for LOM for the refueling of surface vehicles and secondary mining fleet, such as trucks and UVs. There are four tanks on surface with a capacity of 83 m^3 each, giving a total storage of 332 m^3.

Once the underground facilities are commissioned at the main workshop, diesel will be piped down, through a dedicated borehole, from surface. The pipe column to the main workshop fuel storage area will be an "energy dissipation" pipe to prevent high-flow velocities and pressure build-up. The diesel will be batch fed to the underground storage tanks at the main workshop. Fuel will be batch pumped from the tanks at the main workshop to the tanks at the Centrale workshop via a 50 mm diesel pipe. Refueling stations will be available at both the main and Centrale workshops. Fuel will be distributed to working sections from the Centrale and main refueling stations via diesel bowser cassettes. These cassettes will be used to refuel slow-moving or captive equipment such as drill jumbos, bolters, pillar scrapers, and shotcrete spray units. Tier 2 diesel engines will be used, as 15 ppm ultra-low sulfur diesel is unavailable; 50 ppm diesel fuel is currently available on site.

Lubrication oil will be stored in bulk tanks on surface and dispensed to the surface and underground workshops. Bulk lubes will be transferred to surface workshops via dedicated pipelines and transported underground in lubrication cassettes. The underground lubrication cassettes will have dedicated storage areas. Surface storage tanks were designed with sufficient lube storage capacity to operate the mine for approximately 30 days (per grade of oil).

Waste oil and other fluids will be collected in designated cassettes. The waste oil will be discharged from the collection container into a bulk tank, which, when full, will be transported out of the mine.

Explosives Magazine

Consistent explosive supply and distribution is critical for underground mining; the type, delivery, and storage thereof require special design considerations. The underground mine will use the two-component emulsion system, consisting of a base product and a sensitiser combined at the face.

Each type of explosive will be transported underground separately and via different methods. Class 1 explosives, which include Explosive 1.1B and 1.1D, will be transported underground using purpose-built explosives cassettes. Oxidiser and base emulsion will be piped down from surface into storage tanks underground; distribution into the mining areas will be via emulsion cassettes.

AEL was approached to obtain a cost for a suitable vertical drop system for Kansoko. The proposed system is well understood and in use at various operations in Zambia. Emulsion cassettes will be used to store explosives near working areas or in the Centrale emulsion storage magazine.

Concrete and Shotcrete Facility and Distribution

An LOM concrete and shotcrete batch plant will be built on the surface to deliver wet cementitious product. The product will be pumped from the surface batch plant to the borehole servicing the underground delivery facility. Prior to the borehole delivery system, a transmixer / agicar delivery via the decline portals will be used. A positive displacement pump connected to a pipeline will be used to transfer the cementitious mix from the batch plant to the borehole to underground. The transmixer / agicar truck will take the shotcrete to the shotcrete placer pump truck and will discharge the load into the hopper of the shotcrete placement pump.

Compressed Air System

The compressed air system will not include a mine-wide reticulation system from the main surface facility. Piping will be provided from the compressor station on surface to the surface facilities, main underground workshop, and main refuge station. Permanent compressed air piping will be routed from surface through the decline to the underground workings.

Three 1,700 cfm compressors will be located on surface to supply air to the surface infrastructure and underground areas. The compressor will be connected to the mine’s emergency power supply, ensuring that compressed air is supplied to the main refuge chamber during power outages.

Piping will be installed during the development of the primary declines and will extend into the underground maintenance workshop and into the permanent underground refuge chamber. General use compressed air will not extend to other sections of the mine; however, consideration would be given to piping extended to areas for localised use at facilities along the route if needed.

On-board compressors will be available for utility work requiring compressed air. These will be sized for the equipment they will serve. Underground equipment/facilities that will use on-board compressors include the following:

- Jumbos (development, rock bolting, and cable bolting).
- Mechanic Service Trucks (e.g. lube, fuel, maintenance).
- Explosive Loading Trucks (to clean blast holes).
- Shotcrete Placing Equipment.

Water Management

Water management from surface to underground will consist of managing service, potable, and fire water systems. The water management from underground to surface will consist of the mine dewatering system, which includes the production return water system and main pump stations.

Potable water required for use by underground personnel will be provided from the potable water surface supply. The potable water storage will supply the surface facilities as well as the underground mine. A potable water pipeline will be routed from surface through the decline to the underground infrastructure. Pressure reducing stations will be installed as required. Potable water will be provided only for drinking and hygiene purposes and not for any other use.

Mine Service Water (MSW) is necessary for underground operations for drilling, muck pile wet down, wash bays, and dust control. An MSW pipeline will be routed from the surface through the decline to the underground, and to future development, and will remain for later use. Similarly, MSW pipelines will be installed in mine openings driven in ore for panel development and will be left for ore production needs. These pipelines will be removed from each panel for re-use after production is completed. Pressure regulators will manage the increase in static water pressure created as the declines progress. Pressure-reducing stations will have redundant regulators for service and maintenance should issues occur.

Fire Protection System

Several fire suppression system types have been designed and catered for in the operating mine. In all instances, the systems comply strictly with the applicable codes of practice, both local and international. Each system has been designed as a fit-for-purpose solution, which protects the equipment and personnel without restricting operation. On-board foam-based fire suppression systems will be supplied and fitted by the mobile fleet OEM.

Fire water will be drawn from a dedicated source on surface and fed to the underground reticulation system, ensuring the availability of 675 m^3 at all times (size based on other projects with similar underground infrastructure layouts). The reservoir will be constructed with an internal division, subdividing the tank into two equal sections, thus guaranteeing at least 50% availability at all times during maintenance and/or mechanical damage. The tanks will be fitted with a dual suction. The tanks are designed to supply dedicated firewater for a duration of 90 minutes at maximum flow.

Fire water reticulation pipe work will be SANS 62 MED WT galvanised and banded pipe. All fittings and flanges will be Class 16. All isolation / section valves will be Underwriters Laboratories (UL) listed / Factory Mutual (FM) approved. The pipe will be installed in the conveyor declines and has been included in the conveyor designs and costs. The conveyors located in the main decline are installed in the intake and will be equipped with a fire line across the entire length of the belts.

Fire hydrants will be fed off the fire water column and will be placed no further than 60 m apart in the required areas.

All substations and MCCs will have both a smoke detection system in the room as well as a VESDA (Very Early Smoke Detection Apparatus) in the cabinets. Each installation will have its own panel for remote status monitoring via potential free contacts. Each installation will be zoned accordingly, requiring a double knock (two adjacent zones) in simultaneous fire condition prior to the discharge of the gaseous suppression system, thus preventing the possibility of accidental discharge. Each panel will also contain potential free contacts used for the shutting down of associated equipment (e.g. main incomer, air conditioning system).

Flame detectors will be placed at strategic locations and will detect a moving fire in its incipient stage. The detection system will initiate belt shutdown and will activate the solenoid on the associated deluge valve. Each detection system will have its own control panel with potential free contacts for belt shutdown as well as remote monitoring of fire and fault signals.

Materials Handling Logistics and Storage

Materials, equipment, and mining supply items will be delivered by road to the mine site warehouse located at the surface. The mine site warehouse will manage and source services for both the process plant and mining operations.

Designated underground storage areas will be located throughout the mine and typically in proximity to the point of use. Storage areas designated for infrastructure support (e.g. explosive magazines, fuel and lube, warehouse items in transit) will have permanent ground support including shotcrete. These areas will have concrete floors and lighting.

Mining supplies will be managed and sourced from the surface mine site warehouse and will be kept in laydown areas close to the mining operations. The main laydown area will be designed as a drive-thru.

Refuge Stations

Refuge stations will be required to house underground mining personnel in a secure, hazard-free location during emergency conditions. A constructed or modular-style refuge station will be located near the underground maintenance workshop area. If the workshop refuge station is not serviced with suitable compressed air from surface, it will be equipped with self-contained breathing systems. Portable refuge stations will be used to maintain compliance as the mining development faces advance. In the event of an emergency, a notification system, with backup, will signal all personnel to stop work and proceed to the nearest refuge station. All refuge stations will be sized to meet the capacity requirements for the area.

Toilet System

Underground sewerage will comprise two systems: fixed flushing toilets at the main workshops and mobile flushing (non-chemical) toilets for the remainder of underground workings. The mobile toilets are designed as utility vehicle attachments and are easily maneuverable. Each unit will be fitted with a sump and pumped empty into a sewage tank mounted on a cassette carrier. This tank will be transported to surface and emptied into a sewerage disposal system on surface.

Power and Communication Systems

Electrical Substations and Power Distribution

Power will be distributed at 11 kV to the underground mine switchgear from two surface feeder breakers, for redundancy. The underground switchgear will be contained in an E-room and will have separate feeder breakers feeding major mine areas for isolation purposes and to minimise large connected loads to each feeder. Each feeder will feed multiple mine power centres, which will step down the voltage to 525 V for centralised operational loads:

- Mine Medium-Voltage Distribution: 11 kV.
- Secondary Distribution: 525 V.
- Low-Voltage Distribution: 400/230 V.
- Frequency: 50 cycles per second.

The power will be spread to Centrale North, Centrale South, and Sud through the primary development headings. The power will feed the main fixed equipment, such as the conveyors, and the production panels.

Surface loads consists of surface production fans and the cooling plants for the underground mine.

Communication, Controls, and Automation Systems

The backbone for the communications system is based on a redundant fiber network. This system will be used to support all voice and data communication requirements for the Project. Radio communications for the mine will be provided over a leaky feeder system, which will be distributed throughout the entire mine for communication purposes, incorporating hand-held and fixed radios. This will be used to support the people detection system (PDS) / vehicle detection system (VDS), and ventilation-monitoring systems. The leaky feeder can also be used for central blasting.

The mining control will be located on surface in the main surface office, for control of daily mining operations on surface and underground. The equipment provided within these facilities is detailed in the control and instrumentation design criteria. Cameras will be installed at each rock breaker, conveyor transfer point, and pump station. Fiber will be installed for monitoring of the power system and control for conveyors, pumps, and rock breakers. A fiber allowance has been made for ventilation-on-demand, if required.

Upon entering the mine site through the surface access-controlled complex, mining personnel will proceed to the Change House and Lamp Room. Access into and out of the mine will be controlled by means of an electronic tag-in / tag-out system integrated into the cap lamps, which is monitored in the Control Room.

16.2.2.6 Mining Equipment

Criteria considered in equipment selection include suitability, equipment standardisation, 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 equipment requirements are split into two categories: mobile and fixed. The equipment requirements for each category are estimated at a prefeasibility level of accuracy and cover the major components for the operation.

All fixed and mobile equipment used for development and production activities will be based on a 6 Mtpa ore production and associated development. The schedule for equipment purchases and replacement will be based on a rebuild and replace cycle. No equipment will be replaced within 2 years of the end of the LOM.

Mobile Equipment

The average primary mobile equipment fleet is based on specific work activities per the mine schedule. Equipment types—standard profile versus low profile—will vary based upon the areas mined in any given year.

The secondary mobile equipment fleet is based on previous study experience for this and other projects, including the following underground mobile equipment:

- Light Vehicles (manpower).
- Utility Cassette Transports with Cassettes.
- Graders.
- Skid-Steer Cleanup LHD.
- Portable Welder Trailer.
- Concrete Pump Trailer.
- Explosives Loading Trucks.
- Shotcrete Sprayer.

The rebuild and replacement of equipment is calculated based on the life, during operating hours, of an individual piece of equipment. Equipment life is calculated using operating hours as well as vendor-provided actual operating hours for similar operations. Adjustments between engine (diesel) and electrical (percussion for drilling equipment) hours are segregated. The mobile equipment is listed in Table 16.23.

Table 16.23 Mobile Equipment List

Description	Yearly Max. Req.	Purchase/Replace	Rebuild
Development and Production Equipment			
Double-Boom Drill Rig – Standard Profile	22	66	52
Double-Boom Drill Rig – Low Profile	11	31	30
Cable Bolter	3	6	3
LHD – 21 tonne	19	55	20
LHD – 14 tonne	12	21	15
Haul Truck – 63 tonne	19	15	9
Haul Truck – 51 tonne	11	10	3
Mining Scaler	5	8	4
Explosives Loading Truck	9	35	N/A
Mine Support Equipment			
Explosives Transport Truck	5	19	N/A
Concrete / Shotcrete Mixer Truck	17	65	N/A
Telescopic Materials Handler	6	41	N/A
Skid Steer	10	37	N/A
Road Grader	3	8	N/A
Utility Vehicle (UV)	26	136	N/A
Light Duty Vehicle (LDV)	34	215	N/A
Personnel Transport Vehicle – Large	5	20	N/A

Fixed Equipment

Major fixed equipment is defined and addressed within the construction items where they are used, based on the mechanical equipment list. Minor fixed equipment (e.g. drift fans, face pumps, safety equipment) is included as an individual line in the Owner's costs. Table 16.24 summarises the main fixed equipment for the Kamoa mine design.

Table 16.24 Fixed Equipment

Description	Item Qty	Description	Item Qty
Materials Handling		**Electrical and Communications**	
Surface Transfer Tower	1	Main Substation	
Surface Shuttle Conveyor	1	MLCs	
Silo Mechanical	2	Leaky Feeder System	
Tips for Sacrificial Belts	2	**Safety and Miscellaneous**	
Conveyors	30	UG Safety Equipment	
Rock Breakers and Tips	29	Portable Refuge Chambers	
Ventilation		**Surface Facilities**	
Main Fans	10	Fuel and Lubrication Facility	1
Development Fan	87	Concrete / Shotcrete Facility	1
Development / Production Fan	44	Temporary Emulsion Storage Facility	1
GZRM Skid Production Fan	22	Permanent Emulsion Storage Facility	1
Air Doors – Pair	4	**Underground Facilities**	
Mine Air-Cooling Facilities (4 MW, 10 MW)	2	Main Workshop Mechanical and Tools	1
Mine Service Water		Centrale Workshop Mechanical and Tools	1
Metso Pumps	7	Satellite Shop Jib Cranes / Fire Doors	3
Mine Dewatering		Emulsion Storage Facility	1
Portable (Sump) Pumps	72	Concrete / Shotcrete Facility	1
Metso Pump (250 kW)	15	Fuel and Lubrication Facility	1
Pump Skids	109		

16.2.2.7 Personnel

Personnel requirements were developed to support planned development, construction, and operation requirements for the mine. Only personnel directly linked to the operation of the mine are included in this section. Personnel that share other Project activities (e.g. accounting, training, personnel management, environmental, permitting, housing, security, ambulance) are covered in other areas of this report. Personnel requirements are not determined for the following factored personnel:

- Owner’s Project Team.
- EPCM Team.

Figure 16.35 illustrates the average annual personnel requirements for the Project over the LOM.

Figure 16.35 Contractor Vs. Owner Personnel Summary

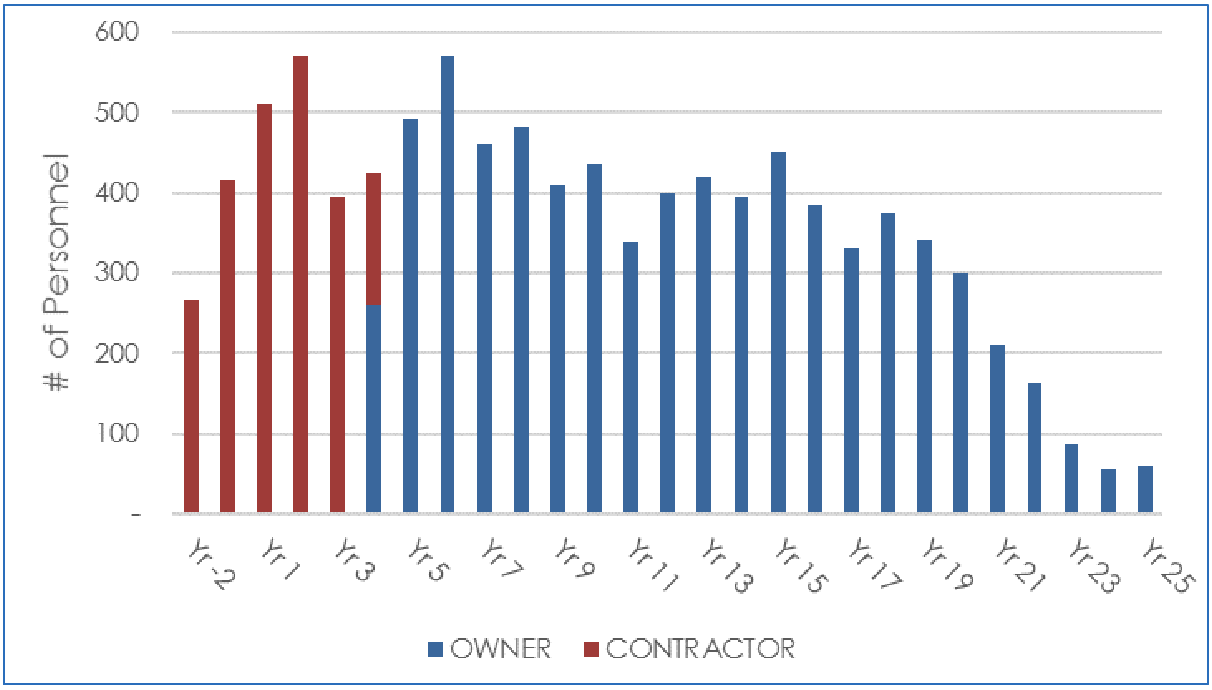


Figure by Stantec, 2017.

Competent mining crews, in particular mobile production equipment operators, are essential in safely achieving production targets. A training department for both mining and engineering has been allowed for in the labour complement. A training facility will be available on surface for technical training. Practical training will be carried out underground, on the job, where final assessment for certification will be done. Recruitment of local labour will require training to be conducted in French and Swahili.

Direct and indirect labor requirements were established to suit the selected mining method, support systems, and general mine requirements during mine development, construction, and operations. Personnel requirements are based on an operating schedule of 12 hours per shift and two shifts per day. Contractor crews will work 360 days per year. Owner capital work and production are accomplished in 360 days per year.

17 RECOVERY METHODS

17.1 Kamoa Concentrator Plant

This section forms part of the Kamoa 2019 PFS, it has not been changed from the Kamoa 2017 Development Plan 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 Kamoa 2017 Development Plan.

The Kamoa 2017 Development Plan is based on the flow sheet envisaged for the Kamoa 2016 PFS, updated to cater for increased throughput in the Kamoa 2017 PFS.

17.1.1 Introduction

This section on recovery methods incorporates assumptions, analysis and findings of the Kamoa 2016 PFS, updated to cater for increased throughput and copper grade.

The Kamoa 2016 PFS process plant consisted of a 3 Mtpa Run-of-Mine (ROM) concentrator incorporating staged crushing, ball mill grinding and flotation. The Kamoa 2017 PFS process plant is based on the same concentrator principle, but at a throughput of 6 Mtpa ROM. The output of the process plant is copper concentrate which is sold.

17.1.2 Process Description

Ore will pass through a 300 mm square grizzly before being conveyed from the mine to surface stockpiles. A diverter is available at the surface to allow waste rock to be stockpiled for removal and to allow stockpiling of ores for later reclaim to the ROM stockpile via an emergency bin. An overbelt magnet removes tramp steel from the ore before it is sent to the ROM stockpile.

Four variable speed apron feeders are available to recover ore from the stockpile and feed to the primary crusher. A second overbelt magnet removes tramp steel from the primary screen feed. ROM ore is fed onto the 50 mm heavy duty primary screen from which the oversize is reports to primary crushing and the undersize reports to secondary crushing.

A variable speed vibrating feeder, located at the base of the primary crusher feed bin, feeds the primary crusher. Primary crushed ore joins secondary crushed ore and is conveyed to the four sizing screen feed bins. Each bin has a variable speed vibrating feeder to feed the respective sizing screen. The screens are double deck with the top deck only working to protect the bottom deck from large particle damage. Oversize from both decks join the undersize from the primary screen and feed secondary crushing. Each of the three secondary crusher feed bins has a vibrating feeder, each feeding a secondary crusher.

Sizing screen undersize is sent to the mill feed stockpile. The undersize has a nominal P_{80} of 8 mm to minimise the potential for scatting (discharging unground oversize) and maximise grinding efficiency in the primary mill. The mill feed stockpile is covered to minimise dust, and has four vibrating feeders below it that feed ore onto the two parallel mill feed conveyors.

Milling is conducted in two identical parallel circuits, each consisting of two identical ball mills in series. A parallel configuration is preferred to manage the mine ramp up period. The primary ball mill is designed to conduct the coarse grind component only and will reduce the ore to a P_{80} of 150 µm. The transfer size between the mills is selected so that all the mills are the same size and draw equivalent power. Final grinding to 53 µm P_{80} occurs in the secondary mill. To maximise grinding efficiency the second mill will use smaller grinding media than the first.

Dry mill feed from the conveyor discharges into the primary mill feed chute, where water is added. The new feed is joined by the primary cyclone underflow. The primary mill discharges through a trommel designed to remove ball scats (spent or broken mill balls) and directs them to a bunker for periodic removal. Mill discharge slurry passes through the trommel to the mill discharge sump. Water is added to control cyclone feed percent solids. Variable speed duty and standby pumps are available to feed the primary cyclone cluster. All the cyclone underflow returns to the primary ball mill whilst the overflow reports to a linear screen. The linear screen removes any tramp oversize from the cyclone overflow to ensure secondary milling efficiency is maximised. The linear screen oversize is scalped on a static screen to remove wood, wire or other material and the undersize slurry gravitates to the secondary mill discharge sump.

New feed to the secondary milling circuit, together with secondary mill discharge, is fed to the secondary cyclone cluster by duty and standby variable speed pumps. Cyclone underflow reports to the secondary ball mill and the primary flotation collector is added to this stream. The secondary ball mill discharges through a trommel screen, to remove ball scats, and the trommel undersize gravitates to the mill discharge sump. The cyclone overflow gravitates to flotation feed conditioning tank.

Roughing and scavenger flotation takes place in two parallel trains. Each train is comprised of a bank of seven cells. The first two cells in the train will perform the roughing duty, with the remainder of the cells performing the scavenging duty. Rougher concentrate from both circuits reports to a common cleaner circuit, whilst scavenger concentrate from both circuits forms part of the common regrind mill feed. Scavenger tails form the majority of the final tails stream.

The flotation feed is pumped from the conditioning tank, via variable speed pumps (two operating, one standby) to the rougher flotation banks. The flotation feed stream is sampled for accounting purposes. Frother and additional collector are added at the feed box; to the first rougher flotation cell, with further addition possible at each subsequent, individual rougher and scavenger cells. Rougher concentrate from the first two cells is pumped, via variable speed pumps (duty and standby) to rougher cleaning cells. Rougher tails feed the first scavenger cell.

Rougher cleaner concentrate is pumped to rougher recleaner flotation and the coarse component of the final concentrate is produced. The rougher recleaner concentrate is pumped to the concentrate thickener. Tails from both the rougher cleaner and rougher recleaners are sent to regrind milling.

The three regrind mill feed streams (scavenger concentrate, rougher cleaner tails and rougher recleaner tails) are pumped to the regrind feed tank. Regrind circuit feed is pumped, via variable speed pumps (duty and standby) to the regrind densifying cyclones. Densifying cyclone overflow reports directly to the regrind product tank and cyclone underflow is fed to the regrind mills. Reground material, with a product P_{80} of 10 µm, reports to the Regrind Product Tank. The reground product is sampled and its particle size is continuously measured, for process control purposes.

Reground material is pumped to the scavenger cleaner flotation conditioning tank (duty and standby pumps). Reagents are added and the slurry is then pumped, via variable speed pumps (duty and standby) to the scavenger cleaner flotation bank. Scavenger cleaner concentrate is pumped to scavenger recleaning and scavenger cleaner tails form part of final tails. The scavenger cleaner concentrate is pumped to scavenger recleaner flotation.

Scavenger recleaner concentrate is pumped to the concentrate thickener feed tank and scavenger recleaner tails are pumped to the final tailings thickener.

The two final concentrate streams are mixed in the concentrate thickener feed tank. Flocculant is added as concentrate-flows to the thickener feed well by gravity. Thickener overflow reports to the concentrate thickener overflow tank, from where it is distributed to the process water circuits. Thickener underflow is pumped (duty and standby) to the filter feed tank and it is sampled for accounting purposes.

All three tailings streams (scavenger tails, scavenger cleaner tails and scavenger recleaner tails) report to the tailings thickener feed tank. Flocculant is added as the slurry flows by gravity to the tailings thickener feed well. All tailings thickener overflow reports by gravity to the process water tank. Tailings thickener underflow is pumped (duty and standby) to the tailings pumping tank and it is sampled for accounting purposes. Multistage slurry pumps pump the slurry to the tailings storage facility.

Concentrate is filtered and then sampled and bagged for transport to customers.

The design criteria are shown in Table 17.1. The two availability figures are in line with industry norms for these types of operations after incorporating allowances for local issues such as power reliability.

Table 17.1 Kamoa Concentrator Plant Design Criteria

	Unit	Value (Design)
Annual Plant Feed	tpa	6,000,000
Overall Crusher Availability	%	75
Crusher Operating Time	hpa	6,570
Crushing Circuit Feed Rate	t/h	913
Overall Mill Availability	%	87
Mill Operating Time	hpa	7,598
Milling Circuit Feed Rate	t/h	790

17.2 Kamoa Concentrator Basis of Design

The concentrator design is based on expectations for the first nine years of operation (Table 17.2). The ROM feed is taken to be 89% Hypogene and 11% Supergene based on the average total production schedule. Appropriate design margins have been incorporated.

Table 17.2 Kamoa Concentrator Basis of Design

Option	Units	Value	Comment
Flotation Feed	Mtpa	6	–
Average Feed Rate	t/h	790	–
Maximum Feed Rate	t/h	869	+10%
Average Feed Grade	% Cu	3.81	Kamoa PFS Mine Plan
Early Ore Grade	% Cu	4.26	Average (Years 1–10)
Design Feed Grade	% Cu	5.14	Max Annual Average Grade Years 1–10 plus 13%
Relative Abundance - Hypogene (%)	Mass %	89	Kamoa PFS Mine Plan
Relative Abundance - Supergene (%)	Mass %	11	Kamoa PFS Mine Plan
Concentrate Grade	% Cu	37	Lower than mine production assumption, see text
Copper Recovery	%	86	From testwork
Design Mass Pull	Mass %	11.9	Based on Design Feed Grade

The ability to blend feed from multiple sources underground should provide the project with a high degree of control over plant feed grade and as such the maximum head grade expected has been chosen to be only marginally higher than the highest annual average grade.

The concentrate grade of 37% Cu was chosen based on a high chalcopyrite feedstock and represents a worst expected case in terms of tonnes to be thickened, filtered and bagged. A grade of 39% Cu has been used in mine planning and this is a legitimate average design grade based on testwork results.

17.2.1 Flow Diagrams

The block flow diagram for the crushing and milling circuit is shown in Figure 17.1.

Figure 17.1 Kansoko Crushing and Milling

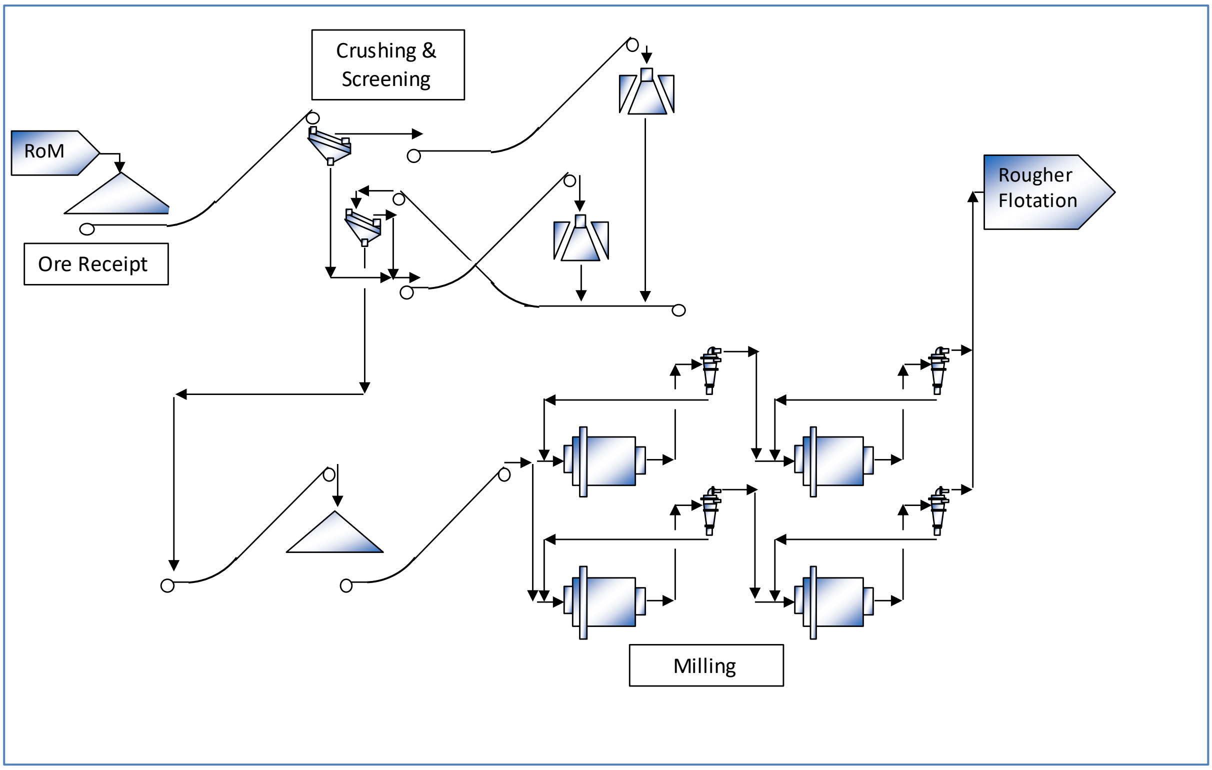


Figure courtesy of MDM, 2017.

The block flow diagram for flotation, concentrate handling and tailings is shown in Figure 17.2.

Figure 17.2 Kansoko Flotation Circuit

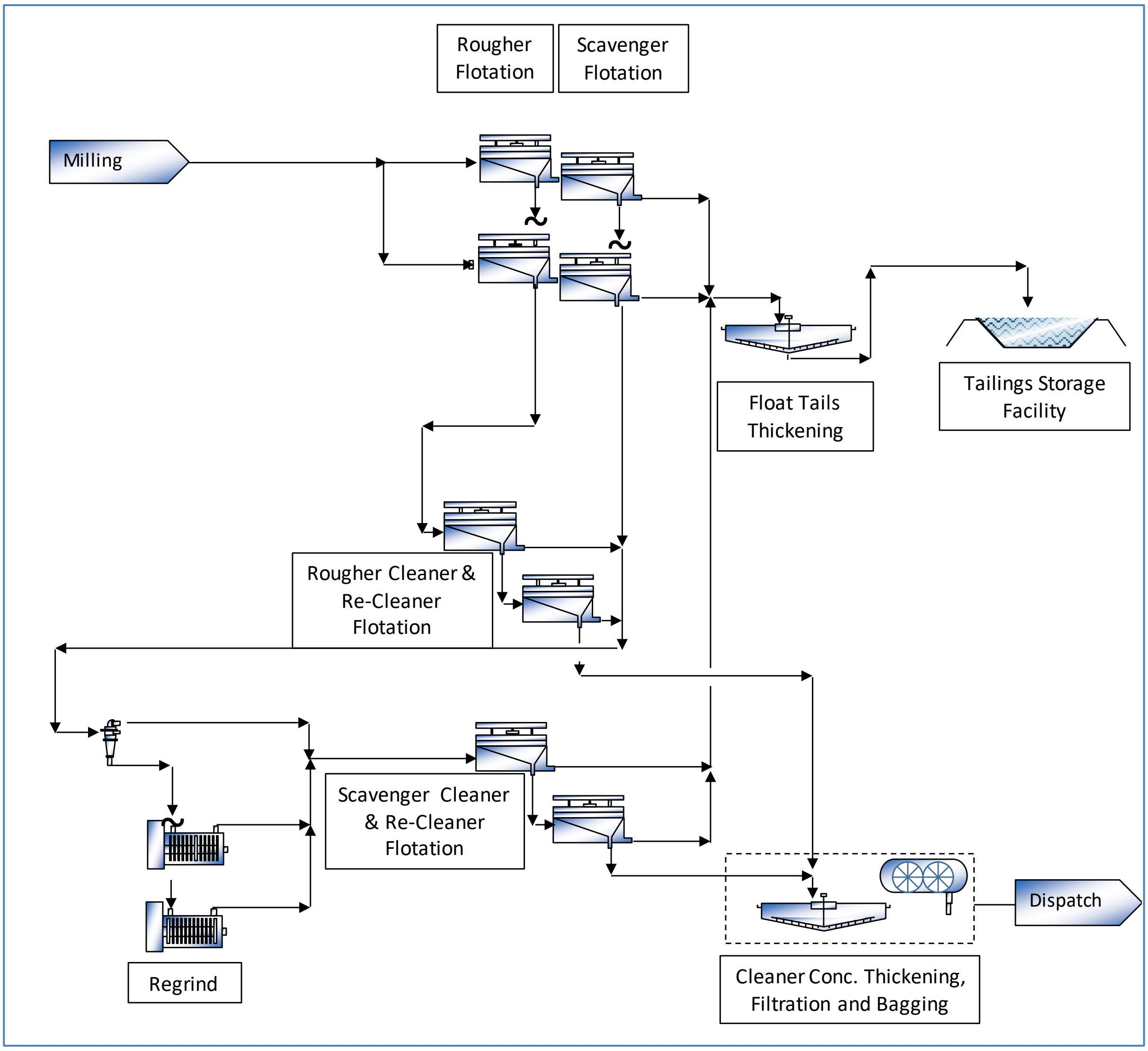


Figure courtesy of MDM, 2017.

17.2.1.1 Reagents, Services and Utilities

Reagent plants, located close to the flotation circuit, provide for the mixing and supply of the necessary reagents for flotation and flocculants for thickening.

All floatation cells are forced air and dedicated blowers supply manifold air for the flotation cells.

Raw water from a wellfield is pumped to a raw water dam. Filtration and treatment plants use the raw water to produce a range of water qualities as required for potable water, gland seal water, fire water and process water usage. Distribution systems for each water type are included, ensuring delivery of sufficient quantity at the required pressure.

Compressed air is supplied and distributed for the use of general plant requirements and filters. A dried air (dew point <0°C) supply is available for air actuated instruments and valves.

17.2.1.2 Concentrator Equipment Specifications and List

Table 17.3 provides a summary of the major mechanical equipment for the proposed concentrator. This list forms the basis of a much more detailed concentrator capital cost estimate.

Table 17.3 Concentrator Equipment Requirements Summary

Item	Description	Size/Capacity	No. Required + standby	Power Installed kW per unit
Crushers	Primary cone	CS660	2	315
	Secondary cone	CH865	3	500
Screens	Primary	2.4 m x 4.27 m	2	45
	Secondary	3.1 m x 6.1 m	4	55
Mills	Primary Ball Mill	22 ft x 36 ft	2	7,000
	Secondary Ball Mill	22 ft x 36 ft	2	7,000
	Concentrate regrind	IsaMill M10000	2	3,000
Cyclones	Primary cluster	750 mm Diameter	4 + 1	500 (1+1 feed pump)
	Secondary cluster	420 mm Diameter	9 + 1	355 (1+1 feed pump)
	Concentrate regrind cluster	165 mm Diameter	15 + 1	75 (1+1 feed pump)
Blowers	Flotation air	65 700 Nm^3/h @ 150 kPa	4 + 1	200
Flotation cells (includes agitators)	Rougher	320 m^3	4	280
	Scavenger	320 m^3	10	280
	Rougher cleaner	50 m^3	5	75
	Rougher recleaner	30 m^3	6	45
	Scavenger cleaner	160 m^3	6	160
	Scavenger recleaner	30 m^3	6	45
Thickeners	Concentrate	20 m Diameter	1	11
	Tailings	50 m Diameter	1	18
Filters	Concentrate	Hyperbaric Disc 57216-2L	2	55
Tailings Pumps	Centrifugal	840 m^3/h	4 + 4	185

Table 17.4 lists the estimated projected water, consumables and power requirements for the concentrator.

Table 17.4 Projected Concentrator Water, Power, and Consumables

Item	Description	Consumption per tonne of Plant Feed	Annual Requirement
Power	Electric	58.8 kWh/t	353 GWh
Water	Raw make-up	0.5 m³/t	3,025 ML
Reagents	Frother	95 g/t	570 t
	Collector	156 g/t	936 t
	Promotor	28 g/t	168 t
	Flocculant (Tailings and Concentrate)	35 g/t	210 t
Consumables	Grinding media (75 mm steel balls)	0.88 kg/t	5,296 t
	Grinding media (35 mm steel balls)	1.34 kg/t	8,034 t
	Grinding media (2 mm Ceramic)	68 g/t	408 t

Most consumables are supplied in bulk bags or containers. A kibble is used to load grinding media into the ball mills. The low abrasion index of the ore (Ai = 0.08) ensures that ball consumption will be relatively low compared to most similar projects.

17.2.1.3 Processing Production Schedule

The processing production schedule is shown in Table 17.5. The schedule is driven more by availability of ore than by plant capacity. The underground mines are being developed in Years –2 to 4, reaching the full 6 Mtpa capacity in Year Four. In Years 1 to 3 the plant will typically require only one line to operate. Alternatively, there may be options for the full plant to run on a reduced daily time schedule.

Table 17.5 Processing Production Schedule

Description	Unit	Total	Project Time (Years)													
			–1	1	2	3	4	5	6	7	8	9	10	11	12	13
Ore Milled	(kt)	125,182	–	1,169	2,608	3,205	4,667	6,014	6,028	6,010	6,015	6,027	6,027	6,027	6,050	6,007
Cu Grade Milled	(% Cu)	3.81	–	3.45	3.99	4.40	4.42	4.26	4.21	4.55	4.39	3.97	3.75	3.85	3.99	3.71
Concentrate Produced	(kt)	11,405	–	97	243	338	498	613	614	679	648	587	558	564	583	539
Concentrate Cu Grade	(% Cu)	37	–	36	37	37	37	37	37	36	36	36	36	36	36	36
Cu in Concentrate	(kt)	4,178	–	35	91	124	183	226	224	245	236	212	201	205	213	196
Cu in Concentrate	(Mlb)	9,211	–	78	200	274	403	498	495	541	520	468	443	452	469	432

Description	Unit	Total	Project Time (Years)													
			14	15	16	17	18	19	20	21	22	23	24	25	26	
Ore Milled	(kt)	125,182	6,077	6,022	6,005	6,019	6,012	6,009	6,018	6,022	4,417	3,126	1,858	943	802	
Cu Grade Milled	(% Cu)	3.81	3.85	3.76	3.46	3.17	3.12	3.24	3.44	3.86	3.49	3.42	3.89	3.83	2.93	
Concentrate Produced	(kt)	11,405	560	544	485	436	431	448	490	551	355	253	168	76	49	
Concentrate Cu Grade	(% Cu)	37	36	36	37	37	37	37	37	37	37	37	37	38	38	
Cu in Concentrate	(kt)	4,178	204	198	178	162	160	168	180	203	132	93	63	29	19	
Cu in Concentrate	(Mlb)	9,211	449	435	392	356	353	369	398	447	290	205	138	64	41	

17.3 Kakula Concentrator Plant

17.3.1 Introduction

This section details the process and engineering design basis of the Kakula Concentrator Plant. The Kakula concentrator process design is based on testwork findings and assessments as presented in Section 13, various trade-off studies and relevant design information.

The design is based on a phased approach of two processing modules, as dictated by the mining ramp-up and production profile. A phased approach further allows for increased processing flexibility and plant redundancy while also reducing the peak capital demand by phasing of capital expenditure.

17.3.2 Kakula Concentrator Basis of Design

The Kakula 2019 PFS design is based on a conventional three-stage crushing circuit (first stage underground), followed by two-stage ball milling and flotation. The Kakula concentrator design criteria are shown in Table 17.6.

Table 17.6 Kakula Concentrator Plant Design Criteria

	Unit	Nominal Value	Design Value
Annual Surface Crushing Circuit Feed	tpa	6,000,000	8,000,000
Surface Crushing Circuit Availability	%	65	72
Surface Crushing Circuit Operating Time	hpa	5,694	6,325
Surface Crushing Circuit Feed Rate	t/h	1,054	1,265
Annual Milling Circuit Feed	tpa	6,000,000	7,600,000
Overall Milling Circuit Availability	%	87	91
Milling Circuit Operating Time	hpa	7,595	7,998
Milling Circuit Feed Rate	t/h	790	950
Milling Module Feed Rate	t/h/module	395	475
ROM Cu Grade	% Cu	6.84	5.48
Final Concentrate Grade	% Cu	57.32	57.32
Mass Pull to Final Concentrate	% Mill Feed	10.2	8.2
Cu Recovery	%	85.5	85.6

17.3.2.1 Crushing and Stockpiling

The Kakula concentrator surface crushing circuit receives crushed ROM from underground mining at a top size of 350 mm. Provision is made in the design however to cater for handling of material at a top size of 550 mm (in the event of not crushing underground, and bypassing of underground grizzly units).

The comminution circuit design is based on the findings from a detailed comminution circuit trade-off evaluation which considered different configurations of the following options:

- Underground jaw crushing followed by Two stage cone crushing (in closed circuit with classification screens) followed by two stage ball milling,
- Closed circuit cone crushing followed by HPGR crushing and two stage ball milling; and
- SABC milling circuit.

The secondary and tertiary cone crusher circuits' sizing has been based on a nominal throughput of 1050 dt/h (maximum 1265dt/h) and a crusher work index of 13.9 kWh/t (85th percentile CWi value from Kakula 2019 PFS, Kamoa Phase 5, and Kamoa Phase 6A testwork). An abrasion index of 0.11 g (85th percentile of Kakula PFS testwork data) was used as design basis.

The mill feed storage system design included a covered stockpile, based on the findings from an evaluation conducted to compare stockpile storage to silo storage. The stockpile live capacity of 10 kt allows for a nominal storage time of just over 12 h of primary mill feed.

17.3.2.2 Milling Circuit

Two separate 3.0 Mtpa milling circuits were selected. The modules are identical in design, and each module comprise of a series primary-secondary, overflow ball milling, each operating in closed circuit with dedicated cyclone clusters.

Each milling circuit is designed to process fresh feed (minus 10 mm) at a nominal throughput of 395 dt/h (design 475 dt/h) to produce a milled product at 80% passing 53 µm, as dictated by the Kakula PFS testwork program conducted at XPS.

Milling circuit simulations were conducted using a combination of comminution parameter testwork on the Kakula material, which was used to derive a breakage function for the Kakula ore. The mills have been sized to cater for the 85th percentile ore hardness as determined from the comminution test data.

The recommended ball mill sizes are 21' Ø x 31½' EGL, for both the primary and secondary mill units. All mills will further be configured with overflow discharge liner arrangements and 7.8 MW (2 x 3.9 MW) variable speed geared pinion drives. The design has been based on a maximum ball charge of 38%. The ball charge, together with the variable speed motor capability, allows for variation in the feed material, required grind size and mill throughput.

17.3.2.3 Flotation and Concentrate Regrind Circuit

As per the milling circuit, the flotation circuit design was based on two separate 3.0 Mtpa modules – as far as practical. The Kakula flotation circuit design is based on the Kakula flow sheet as developed by XPS during the 2017–2018 Kakula PFS testwork campaign.

The flotation circuit residence times have been based on the bench scale flotation circuit residence times. A residence time scale-up factor of 2.1 was applied to determine the design residence time requirements for full-scale operations. The flotation pumping systems and flotation circuits were designed based on the maximum expected mass pulls achieved during the bench scale flotation testwork as detailed in the process design criteria. A froth factor of three was incorporated in concentrate sump and pump sizes.

The concentrate regrind circuit design is based on the outcome of the technology trade-off evaluation which compared a number of fine grinding technologies available in the market. The regrind circuit is designed to produce a slurry at P_{80} 10 µm, from a scavenger concentrate product at F_{80} 17 µm.

17.3.2.4 Concentrate Handling

The concentrate thickening circuit design was based on a common high rate concentrate thickener to treat the combined concentrate from both the 3.0 Mtpa flotation modules.

The concentrate thickener is sized for a maximum tonnage of 79.0 dtph (10% mass pull on 6 Mtpa) and a unit area thickening rate of 0.25 t/h/m^2, as per the Outotec testwork detailed in Section 13. The concentrate thickener design targets a thickener underflow density of 60% solids (w/w).

The concentrate filtration circuit design allows for a phased installation of two horizontal plate filter press units - a single unit is required for the first 3.0 Mtpa, with the second unit only required for processing capacity higher than 3.0 Mtpa. The concentrate filtration circuit is sized for an availability of 85% and a maximum tonnage of 79.0 dtph. A filtration rate of 840 dkg/h/m^2, as dictated by the Outotec testwork, was applied. A filter cake moisture of 8% is targeted.

17.3.2.5 Tailings Handling

The flotation tailings thickening circuit design was based on two identical high rate thickener units to treat the flotation tailings (scavenger tailings and scavenger cleaner tailings) from each module independently. Each tailings thickener is sized for a maximum tonnage of 446 dtph (6% mass pull on 7.6 Mtpa) and a unit area thickening rate of 0.21 t/h/m^2, as dictated by the SGS testwork detailed in Section 13. The tailings thickener design targets a thickener underflow density of 55% solids (w/w), based on tailings settling results from SGS.

The final tailings disposal system includes the backfill circuit feed system, as well as the final tailings disposal system to the TSF. The thickened tailings underflow products from both flotation tailings thickener units will be combined in a common backfill feed sump, for transfer to the backfill plant. Backfill plant tailings (backfill thickener underflow product) at a design density of 40% solids (w/w) is returned to the final tailings sump, from where it is pumped to the TSF. Three final tailings lines (and associated pump trains) are included to provide operational flexibility and to allow for volumetric changes between operating of 3.0 Mtpa, 6.0 Mtpa, and to allow for flexibility of having the backfill plant running or not running.

The final tailings pumping system design is based on a 7.7 km HDPE pipeline, to a final TSF elevation of 1475 mamsl.

17.3.2.6 Sampling and Analysis

The Kakula concentrator design includes automated samplers on the flotation feed, flotation tailings and concentrate thickener underflow stream for metallurgical accounting purposes. In addition, the design includes online analysis of the final concentrate (high-grade cleaner and scavenger recleaner concentrates) as well as the scavenger recleaner circuit feed stream to allow for real-time analysis of these concentrate grades. A number of process control samples have further been included.

A centralised analytical laboratory is included in the design to service the mine and concentrator.

17.3.3 Process Description

A high level block flow diagram of the Kakula concentrator plant is Figure 17.3.

Figure 17.3 Kakula Block Flow Diagram

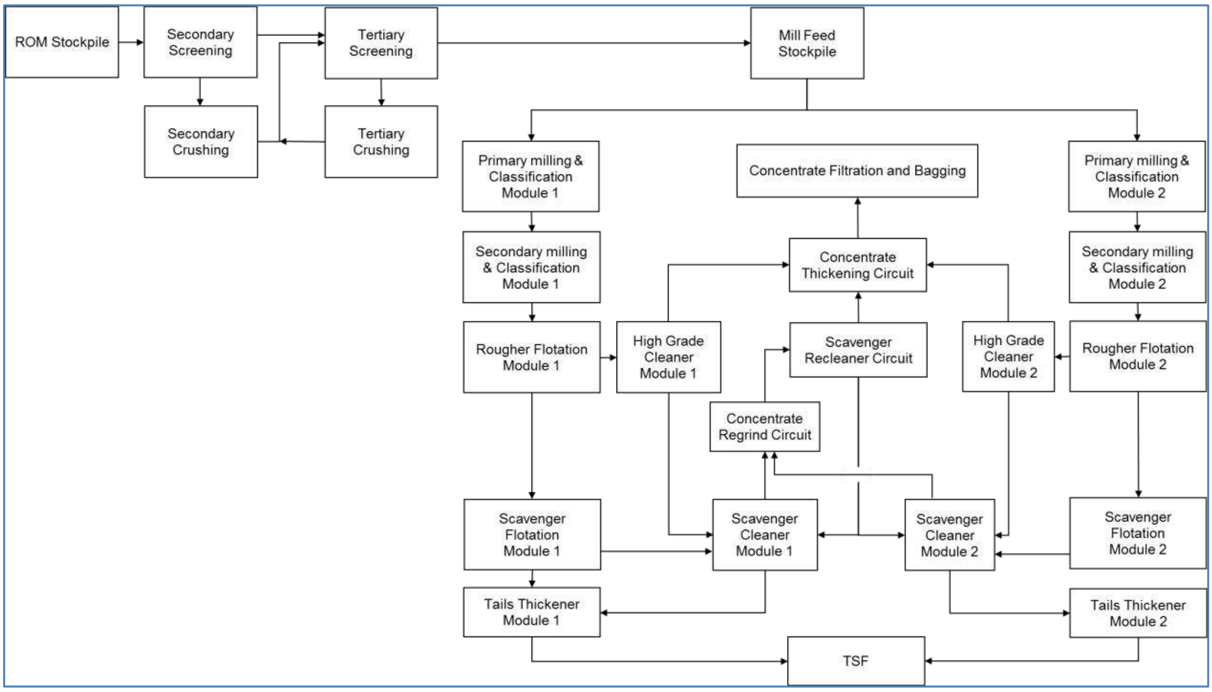


Figure courtesy of DRA, 2019.

17.3.3.1 Run-of-Mine Reclamation

Crushed ROM ore with a top size (F_{100}) of 350 mm from underground, is conveyed to a single 15,000 t ROM stockpile for storage prior to the surface crushing circuit. The material is extracted from the stockpile, at a controlled rate via three variable speed apron feeders, and is discharged onto the secondary screening feed conveyor.

Provision is made for dust control and suitable spillage handling systems, which transfers spillage and run-off to the primary milling circuit.

17.3.3.2 Crushing and Screening

The secondary screening feed conveyor transfers material from the ROM stockpile to the 100 t secondary screening feed bin. The material is screened at 60 mm using a 3.6 x 6.1 m, single deck, vibrating screen. The secondary screen oversize material, roughly 76% of the screen feed, is conveyed to the secondary crushing circuit for size reduction, while the secondary screen undersize material reports to the tertiary screening bin feed conveyor (via a bin and vibrating feeder arrangement).

The secondary screen oversize material reports to the 150 t secondary crushing feed bin, from where the material is extracted at a controlled rate using dedicated feeding systems to feed two continuously operating cone crushers (Model: CS660). Each secondary cone crusher is installed with a 315 kW motor to achieve a size reduction from F_{80} 185 mm to P_{80} 53 mm.

The secondary cone crusher product is conveyed to the 225 t tertiary screening feed bin, together with the secondary screening undersize and tertiary crusher product. The material is extracted from the tertiary screen feed bin at a control rate to feed on to three tertiary screens. Each of the 3.6 m x 7.2 m tertiary screens, screening at 10mm, are equipped with double deck arrangements for deck protection. The tertiary screening undersize product (roughly 43% of the feed) is conveyed to the mill feed stockpile, as a final crushing circuit product. The tertiary screening oversize material is conveyed to the 225 t tertiary crusher feed bin, from where it is fed to three tertiary cone crushers (Model: CH895) at a controlled feed using dedicated feeder arrangements. Each tertiary cone crusher is installed with a 700 kW motor in to achieve a size reduction from F_{80} 48 mm to P_{80} 14 mm. The tertiary crusher product is recycled back to the tertiary screening circuit for classification. The tertiary screening undersize product (P_{80} 7.5 mm) reports to the mil feed stockpile.

Tramp iron removal systems are included on the secondary and tertiary crushing feed conveyors. Provision is made for dust suppression at the screening and crushing buildings, as well as suitable spillage handling systems.

17.3.3.3 Mill Feed Storage

The tertiary screening undersize product is conveyed to the 10,000 t single tunnel covered mill feed stockpile via the mill feed storage conveyor. Four belt feeders operates in a 2-running, 2-standby configuration to transfer material from the mill feed stockpile to each of the primary mill feed systems.

Addition of 70 mm high chrome steel balls onto both the primary mill feed conveyors is achieved by automated ball loading systems. Provision is made for dust suppression at the stockpile reclaim areas, as well as a spillage handling systems. Allowance for scats reloading onto the primary mill feed conveyors is further included in the design.

17.3.3.4 Primary Milling

The primary milling circuit will consist of two identical modules. Each module comprise of a 21'Ø x 31.5' EGL, overflow discharge ball mill (installed with a 7800 kW VSDs) operating in closed circuit with a cyclone cluster consisting of 10 x 500 mm diameter units. Crushed ore at P_{80} 7.5 mm is conveyed from the mill feed stockpile to the primary mill feed hopper where it combines with the primary mill classification cyclone underflow. The primary mill slurry gravitates to the 100 m^3 primary mill discharge sump, via a single deck vibrating scats removal screen, from where it is pumped to the primary mill classification cyclone at a controlled rate and density, using variable speed duty/standby pumping systems. The primary mill classification cyclone overflow product, P_{80} 145 µm, reports to the secondary mill discharge sump as feed.

Spillage produced in the primary mill circuit will report to spillage collection sumps, from where it is pumped to the primary mill discharge sump. Oversize material from the primary mill discharge screen (scats) reports to dedicated scats conveying systems.

The design allows for dedicated mill relining machines to each primary mill, as well as electrical mill feed hopper winch systems to facilitate with maintenance.

17.3.3.5 Secondary Milling

As per the primary milling circuit design, the secondary milling circuit consist of two identical modules. Each module comprise of 21'Ø x 31.5' EGL overflow discharge ball mills (installed with a 7800 kW VSDs), operating in closed circuit with a cluster of 8 x 500 mm diameter classification cyclones.

The primary milling classification cyclone overflow products report to the 100 m^3 secondary milling discharge sump in a reversed feed configuration, where it combines with the secondary mill product, prior to being fed to the secondary mill classification cyclone at a controlled rate and density.

The secondary mill classification cyclone underflow product gravitates to the secondary mill feed hopper, while the cyclone overflow product (P_{80} 53 µm) reports to the mechanical agitated, 500 m^3 rougher flotation surge tank via a two stage metal accounting sampling installation. Milling circuit product is pumped to the rougher flotation feed box using variable speed, duty/standby, pumping systems.

Addition of 30 mm high chrome steel balls are affected using a magnet and loading hopper arrangement, to load the media to the secondary mill feed hopper. Dosing of reagents are via dedicated platework units.

Provision is made in the design for spillage collection and pumping systems.

17.3.3.6 Rougher/Scavenger Flotation

The rougher flotation circuit consist of two identical modules, each comprising of a single bank of seven 300 m³ mechanically agitated, forced air flotation tank cells in series, to produce two concentrate products. Space provision was made for an eight cell at the head of the flotation bank, in order to process a total of 7.6 Mtpa when required. Milling circuit product is pumped to the head of the rougher flotation circuit at a controlled rate and density, where frother is dosed.

A high-grade concentrate product is produced from the first two cells, and gravitates to the 22 m³ high-grade rougher concentrate sump, from where it is pumped to the high-grade cleaner flotation cell, using a fixed speed, duty/standby pumping system. Provision is made for dosing of collector, promoter, and frother to the rougher high-grade concentrate sump, to allow for conditioning of the high-grade cleaner feed slurry.

A low-grade concentrate product is produced from the last five (scavenger) cells and gravitates to the 22 m³ low-grade rougher concentrate sump, from where it is pumped to the scavenger cleaner circuit, using a fixed speed, duty/standby pumping system. Provision is made in the design to divert the second cell's concentrate product to either the high or the low-grade product, when required. Provision is made for dosing of collector, promoter, and frother to the third cell's feed box.

The scavenger tailings product gravitates to the 25 m³ rougher tailings sump, via a two-stage sampling system, before being pumped to a dedicated tailings thickener using a fixed speed, duty/standby pump system.

Spray water, in the form of concentrate thickener overflow effluent, is routed to each of the flotation cell concentrate collection launders to assist with froth transfer.

The design includes multiple spillage collection sumps, equipped with vertical spindle pumps, to transfer spillage to the head of the rougher circuit for re-floating. Emergency showers are included in strategic areas.

17.3.3.7 High-grade Cleaner Flotation

The high-grade cleaner flotation circuit consist of two identical modules, comprising of a single low entrainment Jameson flotation cell, to produce the final high-grade concentrate product.

The high-grade cleaner concentrate gravitates to the 5 m³ high-grade cleaner concentrate sump, from where it is pumped to the concentrate thickening circuit, via a fixed speed, duty/standby pumping system. The design includes an onstream analyser for monitoring of the high-grade concentrate grade. Provision is made for froth washing water to the Jameson cell in the form of raw water.

The tailings from the high-grade cleaner cell gravitates to the 15 m³ high-grade cleaner tails sump, from where it is pumped to the head of the scavenger cleaner circuit, at a controlled rate, using a variable speed duty/standby pump system.

Spillage produced in the high-grade cleaner area is collected in a dedicated spillage sump and pumped to the high-grade cleaner tailings sump via a vertical spindle pump.

17.3.3.8 Scavenger Cleaner Flotation

The scavenger cleaner flotation circuit consists of two identical modules, each comprising of a single bank of 6 x 160m^3 mechanically agitated forced air flotation tank cells in series.

The scavenger cleaner feed consists of the low-grade rougher/scavenger concentrate, together with the high-grade cleaner tailings, and an option to include the scavenger recleaner tailings stream. The design allows for the scavenger recleaner tails to be operated in closed or open circuit – when operated in open circuit the scavenger recleaner tails will report to the scavenger cleaner tails sump. Provision is made for dosing of collector, promoter, and frother to scavenger cleaner feed box. Further, provision is made for spray water to each of the flotation cell concentrate collection launders to assist with froth transfer.

A single concentrate product is produced by the scavenger cleaner circuit, which gravitates to the 20 m^3 scavenger cleaner concentrate sump, where it combines with the scavenger cleaner concentrate product from the second module, prior to being pumped to the concentrate regrind classification cyclone.

The scavenger cleaner tailings gravitates to the 45 m^3 scavenger cleaner tailings sump via a two-stage sampling system, from where it is pumped to the tailings thickener, to combine with the scavenger tailings product from the same module.

Scavenger cleaner area spillage gravitates to the spillage sump from where it is pumped back to the head of the scavenger cleaner flotation bank for cleaning, using a vertical spindle pump. Emergency showers are included in strategic areas.

17.3.3.9 Concentrate Regrind Milling

The concentrate regrind milling circuit consist of a single module, comprising three high intensity 355 kW SMD regrind mills, operating in open circuit with a cluster of 8 x 250 mm diameter cyclones.

The scavenger cleaner concentrate products (P_{80} 17 µm) from both flotation modules are pumped at a controlled rate and density, using a variable speed duty/standby pumping system, to the regrind classification cyclone cluster. The cyclone is designed to target an overflow product of P_{80} 10 µm, which bypasses the regrind mills directly to the 20 m^3 regrind mill product sump. The cyclone underflow product (P_{80} 65µm) gravitates to an equal flow splitter box where it is split as feed to each of the three regrind mills for regrinding to produce a product at 80% passing 10 µm. The regrind mill slurry product combines with the cyclone overflow stream in the regrind mill product sump, from where it is pumped to the scavenger recleaner flotation cell using a fixed speed, duty/standby pumping system. Online grade measurement is provided on the scavenger recleaner feed stream for process control purposes.

Provision is made for a spillage collection sump, complete with a vertical spindle pump to transfer spillage to the regrind mill feed splitter box. Grinding media addition and reclaim systems are further included for each of the three regrind mills.

17.3.3.10 Scavenger Recleaner Flotation

The scavenger recleaner flotation circuit is shared between the two flotation modules and consist of a single low entrainment Jameson flotation cell.

The concentrate regrind circuit product is pumped to the scavenger recleaner cell feed box, where it is combined with the required collector, promoter and frother, prior to final upgrading.

The final medium grade concentrate product gravitates to the 10 m^3 scavenger recleaner concentrate sump from where it is pumped to the concentrate thickening circuit via an onstream analyser.

The tailings from the scavenger recleaner cell gravitates to the 25 m^3 scavenger recleaner tails sump, from where it is pumped to either the scavenger cleaner circuit, or to the final tailings handling circuit (via the scavenger cleaner tailings sump). The scavenger recleaner tailings are transferred using a pumping system operating with two duty pumps and a shared standby pump, to split the tailings product between the two scavenger cleaner flotation modules.

Scavenger recleaner area spillage gravitates to the spillage sump from where it is pumped back to feed of the scavenger recleaner cell using a vertical spindle pump.

17.3.3.11 Flotation Tailings Thickening

The flotation tailings from each module is pumped to dedicated, 55m diameter, high rate thickener units.

The scavenger tailings together with the scavenger cleaner tailings from each module report to their respective tailings thickener feed box, where it is combined with flocculant and coagulant, before gravitating to the thickener feedwell. The thickener systems allow for automatic internal dilution systems.

The tailings is thickened to an underflow product containing 55% solids (w/w), before being pumped to the final tailings handling area, via a two-stage sampling system, using variable speed, duty/standby pumps.

Both tailings thickener overflow products gravitate to a common 5,000 m^3 process water tank for reuse as process water.

Spillage produced in the tailings thickening area gravitate to spillage collection sumps from where it is pumped to the respective thickener feed boxes using submersible pumps.

17.3.3.12 Backfill Feed System and Final Tailings Disposal

Thickened flotation tailings, from both tailings thickener units, are pumped to a common two-stage metal accounting sampling system before gravitating to the 100 m^3 backfill feed sump, via the final tailings splitter box. The thickened flotation tailings are pumped to the backfill circuit using a variable speed pumping system consisting of two duty pumps and pipelines, and a common standby pump.

The backfill plant tailings product is pumped back to the 100 m^3 final tailings sump, using two duty pumps and pipelines supported by a common standby pump, from where it is pumped to the tailings storage facility by means of the final tailings disposal system. The final tailings disposal system consist of three pump trains each comprising three high pressure centrifugal pumps in series, delivering slurry to the TSF via three HDPE pipelines.

In the event that the backfill plant is not operating, the thickened flotation tailings will report to the final tailings sump via the splitter box. Generally, one pump train and pipeline is required for processing of 3.0 Mtpa, with a second pump train and pipeline required after ramp-up to 6.0 Mtpa. The third pump train and pipeline serve as a common standby system under normal operations – however, when the backfill plant is not operating, this system is required to run.

Due to the high operating pressure of the final tailings disposal pump system, the design caters for a dedicated high pressure gland seal water system, consisting of a dedicated storage tank and duty/standby, variable speed multistage pumping system.

The high pressure tailings system valves are operated by a dedicated hydraulic system. Spillage produced in the tailings disposal area gravitate to the spillage collection sump from where it is pumped to the final tailings sump using a submersible pump.

17.3.3.13 Concentrate Thickening

The concentrate products from both flotation modules are thickened in a common 21 m diameter high rate thickener.

The high-grade concentrate products from both flotation modules report to a common single stage vezin, high-grade concentrate sampler. The medium grade concentrate produced by the scavenger recleaner cell is pumped to the single stage vezin, low-grade concentrate sampler. The sampled streams gravitate to the concentrate thickener feed box via a linear trash removal screen where it is combined with flocculant at a controlled rate. Trash removed by the linear screen gravitate to a trash bin for further handling.

The concentrate thickener unit design provides for automatic internal dilution of the feed slurry. The concentrate is dewatered to a pulp containing 60% solids (w/w) prior to being pumped to the filtration area storage area using a variable speed, duty/standby peristaltic pump installation.

The concentrate thickener overflow gravitates to the 150 m^3 concentrate thickener effluent collection tank from where it is reused as flotation spray water.

Spillage produced in the concentrate thickening area gravitate to a spillage collection sump from where it is pumped to the concentrate thickener feed box using a submersible pump.

17.3.3.14 Concentrate Filtration Feed

The thickened concentrate is pumped to either one of two 500 m^3 mechanically agitated filtration feed tanks, via a single stage metal accounting sampling system and splitter box.

The thickened concentrate is fed to either one of two, 60 m^2, horizontal plate pressure filters using two duty pumps supported by a common standby pump.

Spillage produced in the filtration feed area gravitate to a spillage collection sump from where it is pumped to the concentrate filtration feed tank splitter box using a vertical spindle pump.

17.3.3.15 Concentrate Filtration

The thickened concentrate is dewatered to a filter cake at a target moisture of 8.0% solids (w/w).

The filter cake product reports dedicated filter cake discharge conveyors, which in turn either transfer the filter cake to the concentrate loadout conveyor, or to emergency stockpiles. Filter effluent report to the concentrate thickening circuit via a spillage system.

Vendor supplied auxiliary systems to the filter press units include hydraulic pressing systems, cloth wash systems, manifold wash systems, pressing air and drying air systems. The building design includes an overhead travelling crane for use during maintenance.

17.3.3.16 Concentrate Bagging and Loading

The design of the concentrate bagging and loading facility is based on a 16hour operation per day. Provision is made in the design to load 66% of the produced concentrate into bulk trucks, and the remainder into bags.

The filter cake product from the pressure filters report to the concentrate loadout conveyor by which it is conveyed to the vendor supplied concentrate bagging and loading facility. The concentrate bagging and loading plant consist of three modules, each fed from a dedicated 250 m^3 storage bin and associated rotary screw feeders. Initially, only a single module will be equipped with a bagging carousel however, the design of the other two bulk loading units allows for easy conversion to additional bagging systems.

The two bulk loading systems are designed to fill bulk trucks on a weight basis, after which the loaded trucks are sampled using an auger sampler installation. The bag loading module is equipped with a sampling system to allow for compositing of a concentrate sample on a shift basis, and linking of the analysis to the number of bulk bags filled during that shift.

Spillage produced in the bagging and loading area is washed to a spillage collection sump from where it is pumped to the concentrate thickening circuit for reintroduction to the system using a vertical spindle pump.

The design further caters for cleaning of concentrate from the trucks prior to dispatch in the truck wash area. All effluent produced in the truck wash area is pumped to the concentrate thickening circuit.

17.3.3.17 Air Services

Low pressure blower air to each of the forced air tank flotation cells are supplied by four variable speed multistage centrifugal air blowers operating as three running and one standby. Each blower is equipped with a dedicated suction filter, and suction silencers. In addition to silencers fitted on each suction, each blower is further equipped with dedicated delivery line silencer units.

The design includes seven air compressors, supplied as vendor packages, dedicated to the concentrator plant. The instrument and plant air required on the concentrator plant is produced by three low pressure plant/instrument air compressors (two running, one standby) at 1300 kPa. Plant air is stored in a single 30 m^3 air receiver at 1300 kPa, from where it is distributed across the concentrator at 750 kPa. Plant air is not passed through a drying system. The majority of the air produced by the plant/instrument air compressors passes through two air filtering and drying systems (one system per concentrator module) before being stored in two 10 m^3 instrument air receivers (one per concentrator module). Due to the high instrument air requirement from the flotation circuits, the design further includes for two additional 10 m^3 instrument air receivers – one located in each rougher/scavenger flotation module. As with the plant air, the instrument air is stored at 1300 kPa, distributed at 750 kPa.

Drying air to the concentrate filter units is supplied by two 1300 kPa compressors, in a duty/standby configuration, and stored in a single 10 m^3 drying air receiver from where it is distributed to either one of the filter units. Pressing air to the concentrate filter units is supplied by two 1600 kPa compressors, in a duty/standby configuration, and stored in a single 2 m^3 pressing air receiver from where it is distributed to either one of the filter units.

17.3.3.18 Water Services

The water circuit design for the concentrator circuit consists of three separate systems, i.e. process water, raw water and potable water.

The 5,000 m^3 process water tank is fed by any excess concentrate thickener effluent, TSF return water, both tailings thickener overflow products, treated sewerage and storm water run-off. Process water, required for dilution, is distributed to each concentrator module via a dedicated process water pump, supported by a common standby pump. Process water is further used for general flushing and hosing, supplied by a single flushing and hosing pump. The design further includes a process water bleed pump to transfer any excess process water to the excess water treatment plant.

Raw water supply to the concentrator area is pumped from the main raw water storage dam using a duty/standby pumping system. The raw water is stored locally at the concentrator in the 1,000 m^3 raw water tank, from where it is distributed across the concentrator plant for use as gland seal water, reagent make-up, dust suppression, and clean spray water. The gland seal water circuit is designed as a separated circuit based on pressure requirements, and includes dedicated pressure controlled gland seal water ringmains to each of the milling-flotation modules, fed from dedicated gland seal water pumps. A third pump is provided as a shared standby to the gland seal water circuit. Other raw water is distributed to the required points at a controlled pressure using a duty/standby concentrator raw water pumps.

Potable water from the potable water treatment plant is collected in the 20 m^3 elevated concentrator potable water storage tank, from where it is gravity fed to the required distribution points. Potable water is further used as feed to the 680 m^3 concentrator fire water tank, from where it is distributed across the concentrator circuit at a controlled pressure using a vendor supplied fire water pumping system consisting of an electrical pump, a jockey pump, and a diesel pump.

17.3.3.19 Collector Make-up and Dosing

Sodium IsoButyl Xanthate (SIBX) is used as the main collecting reagent in the flotation circuit. SIBX is delivered in powder form (850 kg bags) and stored in the reagent store. As required, bags are moved from the reagent store to the SIBX make-up area.

During batch make-up, a bag is manually hoisted and discharged into the 20 m^3 mechanically agitated collector mixing tank, where it is diluted with raw water to achieve the targeted dosing strength of 10% (w/v). Once the solution is well blended, and the solution strength confirmed by manually sampling, the solution gravitates to the 30 m^3 collector dosing tank from where it is distributed to the designated dosing points using a duty/standby peristaltic pump system to feed a pressure controlled ringmain.

Spillage produced in the SIBX make-up and dosing area is collected in a dedicated spillage sump and pumped to the final tailings disposal sump via a vertical spindle pump. Provision is made for safety showers in the area.

17.3.3.20 Promoter Make-up and Dosing

AERO 3477 is used as promoter in the flotation circuit, and is delivered as a 50% (w/v) liquid by 30 t bulk tankers.

The design includes for a single promoter offloading pump to transfer the reagent from the bulk tankers to the 30 m^3 promoter receiving tank. The promoter offloading pump is used to transfer the 50% (w/v) solution to the 30 m^3 mechanically agitated promoter mixing tank where it is diluted with raw water to achieve the target dosing strength of 10% (w/v). Once the solution is well blended, and the solution strength confirmed by manually sampling, the solution gravitates to the 10m^3 promoter dosing tank from where it is distributed to the designated dosing points using a duty/standby peristaltic pump system to feed a pressure controlled ringmain.

Spillage produced in the promoter make-up and dosing area is collected in a dedicated spillage sump and pumped to the final tailings disposal sump via a vertical spindle pump. Provision is made for safety showers in the area.

17.3.3.21 Frother Dosing

SF22 is used as frothing agent in the flotation circuit, and is delivered in a concentrated liquid form using 1 t intermediate bulk containers (IBCs). The design allows for dosing of frother directly from these IBCs, without any further dilution, using dedicated variable speed peristaltic pumps.

No additional spillage handling systems are included in the design, as the IBCs are located within the flotation area bunds.

17.3.3.22 Flocculant Make-up and Dosing

The design allows for use of Magnafloc 10 as flocculant at the concentrate and both tailings thickeners, and the use of Magnafloc 380 as coagulant at both tailings thickeners. The coagulant is delivered as a powder in 1 t bags. Identical mixing and dosign systems are provided for each reagent.

For both make-up systems, the bags are manually hoisted and discharged into the vendor supplied flocculant bulk bag bin receiver. A screw feeder is used to transport the dry flocculant into the flocculant wetting tank, prior to being pumped to either one of two 100 m^3 mixing/dosing tanks.

Provision is made for raw water addition to the wetting tank as well as both the mixing/dosing tanks for dilution to a transfer strength of 0.5% (w/v), at which it is pumped to the respective thickening circuits using a duty/standby pump system to feed a pressure controlled ringmain. The flocculant is further diluted to 0.05% (w/v) at the dosing points using inline mixers.

Spillage produced in the flocculant make-up and dosing area is collected in a dedicated spillage sump and pumped to the final tailings disposal sump via a vertical spindle pump.

17.3.4 Concentrator Services Requirements

Table 17.7 lists the estimated projected water, consumables and power requirements for the concentrator.

Table 17.7 Projected Concentrator Water, Power, and Consumables

Item	Description	Consumption per tonne of Plant Feed	Annual Requirement
Power	Electricity	54.8 kWh/t	416 GWh
Water	Raw make-up	0.40 m³/t	2,890 ML
Reagents	Frother	203 g/t	1,256 t
	Collector	310 g/t	2,069 t
	Promotor	56 g/t	675 t
	Flocculant	26 g/t	156 t
	Coagulant	40 g/t	242 t
Consumables	Grinding media (70 mm steel balls)	0.350 kg/t	2,100 t
	Grinding media (30 mm steel balls)	0.600 kg/t	3,600 t
	Grinding media (3 mm Ceramic)	20 g/kWh	35 t

17.3.5 Processing Production Schedule

The processing production schedule is shown in Table 17.8.

Table 17.8 Processing Production Schedule

Description	**Units**	**LOM**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**	**2030**	**2031**	**2032**
Total tons milled	kt	187,048	4,800	6,150	8,302	9,067	9,101	9,058	9,082	9,054	9,036	9,034	9,044	9,073
Module 1	kt	119,728	2,400	3,075	5,302	6,067	6,101	6,058	6,082	6,054	6,036	6,034	6,044	6,073
Module 2	kt	67,320	2,400	3,075	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000
Mill feed grade	% Cu	5.48%	6.73%	7.06%	6.70%	6.90%	6.66%	6.75%	6.66%	6.37%	5.47%	5.18%	5.25%	5.39%
Mass pull to product	% mill feed	8.17%	9.88%	10.61%	9.94%	10.35%	9.99%	10.12%	9.98%	9.54%	8.15%	7.70%	7.81%	8.02%
Concentrate produced	dry kt	9,776	237	326	527	628	609	613	607	578	492	465	472	487
Concentrate grade	% Cu	57.32%	57.32%	57.32%	57.32%	57.32%	57.32%	57.32%	57.32%	57.32%	57.32%	57.32%	57.32%	57.32%
Cu Recovery	%	85.35%	84.21%	86.06%	85.06%	86.00%	85.91%	85.94%	85.90%	85.79%	85.38%	85.24%	85.27%	85.34%
Description	**Units**	**2033**	**2034**	**2035**	**2036**	**2037**	**2038**	**2039**	**2040**	**2041**	**2042**	**2043**	**2044**	**2045**
Total tons milled	kt	9,032	9,005	9,087	9,024	9,036	8,836	8,248	7,639	6,111	3,771	2,761	2,325	1,372
Module 1	kt	6,032	6,005	6,087	6,024	6,036	5,836	5,273	4,939	3,055	1,886	1,381	1,162	686
Module 2	kt	3,000	3,000	3,000	3,000	3,000	3,000	2,975	2,700	3,055	1,886	1,381	1,162	686
Mill feed grade	% Cu	5.23%	5.44%	5.36%	4.96%	4.32%	4.35%	3.86%	4.18%	4.45%	3.89%	3.61%	3.71%	3.86%
Mass pull to product	% mill feed	7.78%	8.10%	7.99%	7.36%	6.39%	6.44%	5.68%	6.18%	6.59%	5.73%	5.30%	5.45%	5.68%
Concentrate produced	dry kt	469	486	486	443	386	376	300	305	201	108	73	63	39
Concentrate grade	% Cu	57.32%	57.32%	57.32%	57.32%	57.32%	57.32%	57.32%	57.32%	57.32%	57.32%	57.32%	57.32%	57.32%
Cu Recovery	%	85.26%	85.37%	85.33%	85.12%	84.76%	84.78%	84.46%	84.67%	84.84%	84.48%	84.29%	84.36%	84.46%

17.4 Comments on Section 17

17.4.1 Kamoa Concentrator Plant

This plant design is based on the flow sheet used in the Kamoa PFS in 2015. The laboratory flotation flow sheet used in the plant design has been applied a number of times since the PFS was completed at two different laboratories and has proven to give acceptable results for a variety of ore types. This recent work has provided confidence that the liberation characteristics of the ores in the Kamoa and Kansoko areas of the copper system are similar and respond well in identical test conditions. In addition, comminution testing shows that ores from all areas have similar breakage characteristics and will respond in a similar fashion during crushing and grinding. Overall, no flow sheet risks arose as a result of testing the various different feeds.

ROM ore is assumed to have a topsize of 300 mm, controlled by intensive blasting and 300 mm square grizzly installations at each ore dump point underground. If this topsize control is found to be unmanageable by blasting alone, then additional underground crushing may be required. Note that underground grizzly sizes can only be relaxed with caution as particles in excess of 300 mm will cause problems for the conveying systems that bring material to the surface from underground.

The plant design is based on a 53 µm flotation feed P_{80} and a 10 µm regrind P_{80} of the flotation middlings. Testing has shown these parameters to be robust and good control has been gained in the more recent testwork over silica content of the final concentrate.

The flotation circuit configuration deliberately avoids recycle streams in accordance with the XPS testing philosophy. This results in (at least theoretically) well-defined residence times throughout the circuit. However, it presents a risk with regard to managing varying ore grade and copper sulphide mineralogy. The most likely stream to be recycled in the current configuration is the scavenger recleaner tail (recycle to scavenger cleaner feed). Another possibility is to recycle rougher recleaner tail to rougher cleaner feed as this may reduce the regrind duty. Flow sheet provision for the scavenger recycle is allowed and consideration should be given to making provision for the rougher recycle.

The copper mineralisation determines how much copper is recoverable by flotation and the grade of concentrate that can be generated. The mineralisation is highly variable and further work is needed to better define mineralogy in the various parts of the Kamoa deposit. Good quality mineralogical information will be necessary for feed grade blending and controlling final concentrate grade. Even higher quality information is required if an on-site smelter is constructed in the future of the project.

In the dominant hypogene ores concentrate grade will vary with the relative chalcopyrite, bornite, chalcocite and digenite proportions and also with the amount of silica recovered to concentrate. One variable, the silica content of the concentrate, is now shown to be controlled in the test procedure. The variable copper mineralogy will also need to be understood and may be able to be inferred from Cu: S ratio. Samples with varying Cu:S and fully defined copper mineralogy must be subjected to variability testing, which is in plan.

Hypogene ore copper recovery is always high as there is little unfloatable copper mineralisation present. Improved hypogene ore definition during planning and production requires a means of identification of the relative proportions of the important copper minerals in core and drill cuttings and this should also be explored in the next study phase.

Supergene ore has broad variability in both the relative proportions of the floating copper minerals (as described for hypogene in the previous paragraph) and the proportion of copper that will be lost to tailings in non-floating minerals. This leads to uncertainty with regard to both copper recovery and grade of copper concentrate. Improved supergene ore definition requires identification of the relative proportions of chalcopyrite, bornite, chalcocite, diginite and covellite (floating copper minerals) together with a measure of how much copper is in non-floating species such as native copper, azurite, malachite and cuprite. The overall project impact of the supergene uncertainty is important, but it is not felt to be material to the project economics. Supergene only represents 11% of the concentrator feed in the Kansoko mine plan and supergene will always be blended with hypogene in the mill feed.

17.4.2 Kakula Concentrator Plant

This plant design is based on the flow sheet developed at XPS during the Kakula PFS campaign, which has proven to give acceptable results for a variety of samples. Testing of the Kakula West and Kamoa material on the Kakula PFS flow sheet has provided confidence that the different deposits targeted can be treated using a common concentrator design. In addition, comminution testing shows that ores from all areas have similar breakage characteristics and will respond in a similar fashion during crushing and grinding. Overall, no flow sheet risks arose as a result of testing the various different feeds.

ROM ore is assumed to have a topsize of 350 mm, controlled by intensive blasting and 350 mm square grizzly installations at each truck tip underground. Flexibility has been included in the design by designing to an maximum expected blasted topsize of 550 mm.

The plant design is based on a 53 µm flotation feed P_{80} and a 10 µm regrind P_{80} of the flotation middlings. Testing has shown these parameters to be robust. The flotation circuit configuration deliberately avoids recycle streams in accordance with the XPS testing philosophy. This results in (at least theoretically) well-defined residence times throughout the circuit. However, it presents a risk with regard to managing varying ore grade and copper sulphide mineralogy. The most likely stream to be recycled in the current configuration is the scavenger recleaner tail (recycle to scavenger cleaner feed). Flow sheet provision for the scavenger recycle is allowed.

18 PROJECT INFRASTUCTURE

18.1 Kakula Site Infrastructure

18.1.1 Introduction

This section describes the project infrastructure work that was developed for the Kakula 2019 PFS. Surface infrastructure, was broken down into the following three principle areas:

- Mining surface infrastructure – all supporting infrastructure located within the mine portal area, i.e. roads, buildings, workshops, dams, services, bulk earthworks and electrical reticulation, etc.
- Process plant infrastructure –all supporting infrastructure located within the process plant area, i.e. roads, buildings, dams, services, bulk earthworks and electrical reticulation, etc.
- General site infrastructure – bulk supply infrastructure (includes water and power) located within the waste management area, general office area and all supporting infrastructure linking the three areas together.

18.1.2 Site Plan and Layout

A plan showing the locations of the mines and key infrastructure for Kakula and Kansoko mines is shown in Figure 18.1.

Figure 18.1 Kamoa-Kakula IDP19 Site Plan

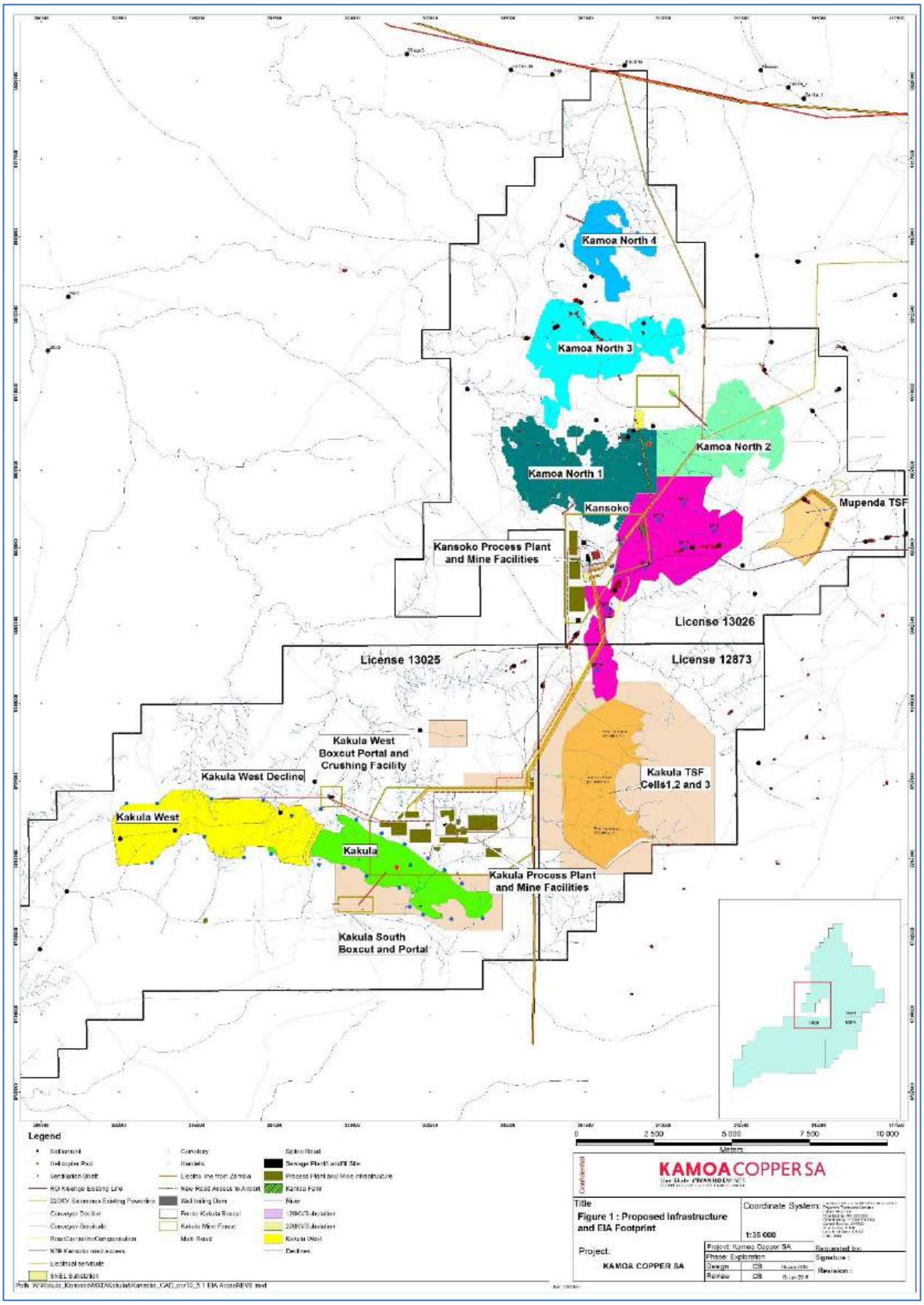


Figure by Kamoa Copper SA, 2019.

18.1.3 Block Plan Development and Layout

Plot and block plans were developed, with a holistic view of the complete mine lease area and using the philosophies agreed upon with the client. Layouts for the mining area, process plant, tailings facility and various general infrastructure sections, were completed with these areas in relation to one another and individually.

All layout design work was undertaken in close co-operation with the Kamoa-Kakula Project owner's team using the recent LIDAR survey that has been done. The following major factors specifically influenced the layout of plot and block plans:

- Site Topography(LIDAR)
- The requirement to allow sufficient footprint for potential expansion.
- The hydrology requirements.
- Existing infrastructure due to early works.
- Process plant requirements.

The process plant will be constructed in two phases with two 3 Mtpa concentrator modules coming on line to meet the staged mining ramp up. This phased approach was considered in the development of the plot plan. The plot plan also incorporates existing infrastructure such as the box-cut, workshops, offices and fence line.

The site topography was considered in the placement of major infrastructure, pipelines, dams and the location of the tailings storage facility.

18.1.4 Electrical, Control and Instrumentation Design

18.1.4.1 Generation

Power for the Kamoa-Kakula Project is planned to be sourced from the DRC's state-owned power company Société Nationale d'Electricité (SNEL), electrical interconnected grid. This electrical grid faces a shortage of power generation due to ageing hydropower plants with a number of non-working turbines that require repair.

The hydro power plants in the SNEL southern grid that are considered in the Ivanhoe SNEL power project are: Koni, Mwadingusha, and Nzilo. All three require refurbishing. The three plants combined could produce over 200 MW. Prior to completion of the refurbishment, development and construction activities at Kamoa will be powered by electricity sourced from the SNEL grid and on-site diesel generators.

In June 2011, Ivanhoe signed a Memorandum of Understanding (2011 MOU) with SNEL. The 2011 MOU led to the signing of a pre-financing agreement with SNEL in June 2012 under which Ivanhoe pledged a loan of USD 4.5 million for the emergency repair of generator unit 1 at Mwadingusha hydroelectric power station. This will unlock 10 MW of power required for development and construction activities.

After subsequent negotiations, SNEL granted Ivanhoe an exclusive right to conduct full rehabilitation on the Mwadingusha and Koni plants following completion of a feasibility report on the work. A study to rehabilitate the Mwadingusha and Koni power plants was carried out by Stucky Ltd in 2013 (Stucky Report on Mwadingusha and Koni).

On 14 March 2014, SNEL and Ivanhoe signed a Financing Agreement for the rehabilitation of the two power stations and associated high voltage infrastructures. This financing agreement is in the form of a loan to SNEL that will be re-paid with 40% discounted power tariffs.

Once completed, the upgrade and modernization of Mwadingusha, is expected to be restored to its installed capacity of about 71 MW. After completion and handover, HPP Mwandingusha will supply electrical energy to the Congo National Grid as well as to the copper mining activities at the Kamoa-Kakula project by Ivanhoe mines.

SNEL guarantees 100 MW to Kamoa once Koni and Mwadingusha is refurbished.

In 2013, Ivanhoe signed an additional Memorandum of Understanding (2013 MOU) with SNEL to upgrade a third hydroelectric power plant, Nzilo 1. A study to rehabilitate the Nzilo 1 power plant was carried out by Stucky Ltd in 2014 (Stucky Report on Nzilo 1). It is proposed to upgrade the Nzilo 1 hydroelectric power plant to its design capacity of 100 MW. The location of the power planst is shown in Figure 18.2.

Figure 18.2 Power Plants Locations

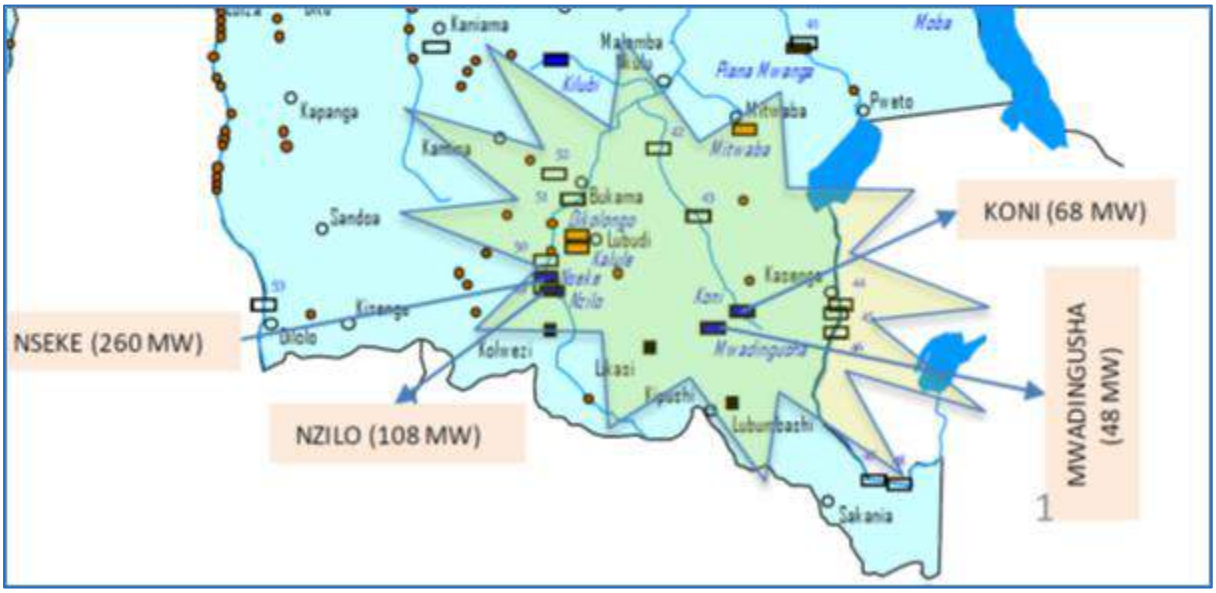


Figure by Ivanhoe, 2017.

Mwadingusha Hydroelectric Power Plant

The Mwadingusha hydro power plant is located on the Lufira River, approximately 70 km from the city of Likasi in the province of Haut-Katanga in the DRC. The hydro facility was built in 1928 and comprises six turbines with an installed generation capacity of 71 MW at a gross hydrostatic head of 114 m. Turbines four and five were installed in 1938, whilst turbine six was installed in 1953. Of the turbines installed, turbines four, five, and six, are currently operational.

Koni Hydroelectric Power Plant

Koni is located 7 km downstream of Mwadingusha and was built in 1946 with an installed generation capacity of 42 MW at a hydrostatic head of 56 m. The turbine hall comprises three turbines, only turbines one and two are currently operational.

Nzilo Hydroelectric Power Plant

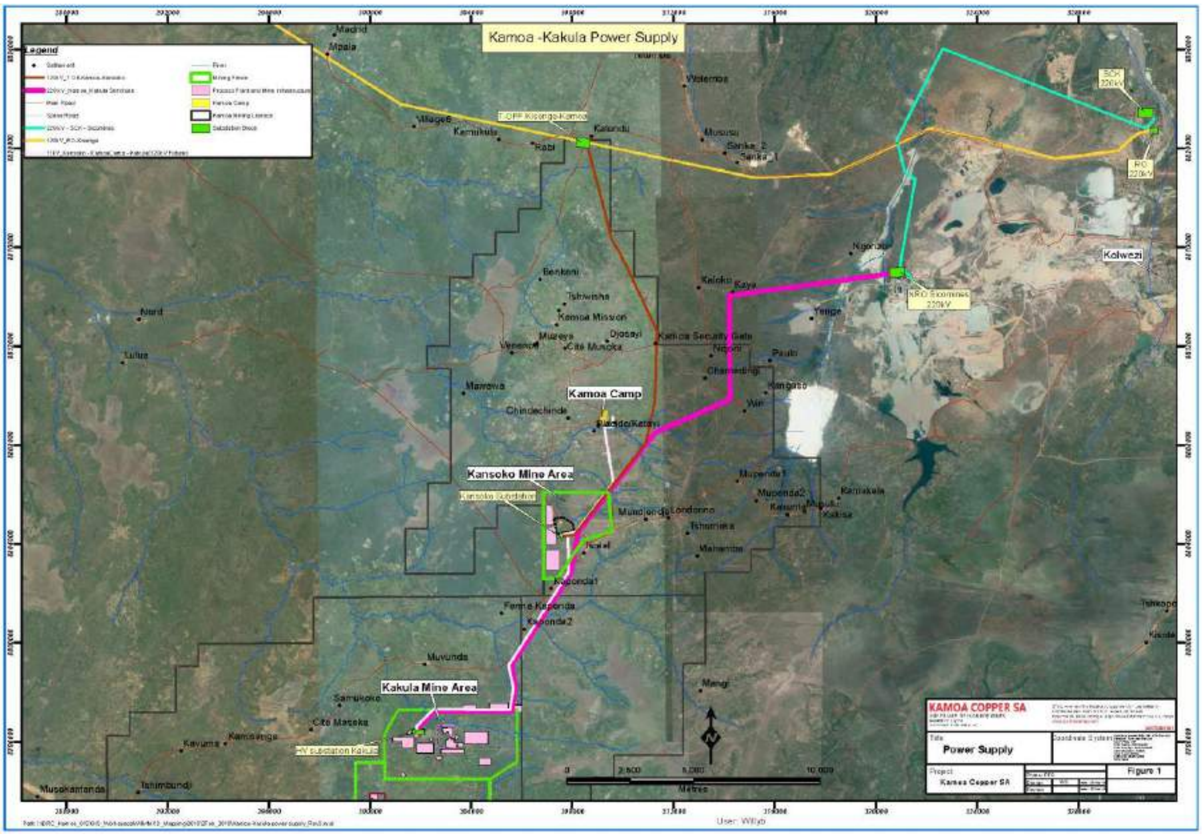
The Nzilo hydro power plant is located on the Lualaba River, approximately 30 km from the city of Kolwezi in the DRC province of Lualaba. The hydro facility was built in 1952 and comprises four turbines with an installed generation capacity of 108 MW at a gross hydrostatic head of 74.5 m. Three out of four turbines installed are currently operational but need to be renewed.

18.1.4.2 Bulk Power Supply

Power to the Kamoa 220 kV substation will be supplied from the new NRO substation (Nouveau Répaprtiteur Ouest). The NRO substation will be financed by Ivanhoe Mines Energy DRC and forms part of the loan agreement. This substation will be supplied from the 220 kV SCK substation in Kolwezi. SCK substation is a major 220 kV transmission station in the SNEL’s southern network and is supplied from the 500 kV Inga – Kolwezi DC lines.

A double circuit 220 kV transmission line (35 km) will be installed between NRO and the Kamoa 220 kV infeed point. Metering will be at the take off point at NRO substation as the line will remain the property of Ivanhoe Mines Energy DRC. Figure 18.3 indicates the planned 220 kV line servitude.

Figure 18.3 Planned Kakula 220 kV Overhead Line



Five 220/22 kV 40 MVA transformers will be installed. The supply is designed to provide N+1 redundancy on both the transmission line and the transformers. Construction of the 220 kV OHL (Overhead Line) and substation is forecast to commence in the second half of 2019. The final budget quotation, self-build and electricity supply agreements have been drafted for inclusion in the capital estimate. Monitoring of the incoming supply is done at the Kamoa 220/22 kV Substation.

18.1.4.3 Predicted Electrical Consumption and Notified Maximum Demand (NMD)

A bottom up estimating methodology was used to arrive at a predicted electrical power consumption and the notified maximum demand (NMD) for the various surface and underground proposed installations. The NMD is the maximum electrical power demand in kVA, over a half hour period.

The complete list of connected requirements by area, are summed in Table 18.1 below to derive a total connected maximum demand load, which are reduced to connect running loads by excluding standby circuits, and further reduced to absorbed power running loads as stipulated by the process and mechanical data.

Table 18.1 Kakula Notified Maximum Demand

	Installed [MW]	Running [MW]
	159.20	116.76
Concentrator Plant (3000)		
3 Mtpa Plant (Phase 1)	37.05	26.85
6 Mtpa Plant (Phase 2)	30.53	21.33
Subtotal [MW]	67.58	48.17
Mining (Surface & Underground)		
Paste-Plant	12.00	9.40
Mining Crew	10.35	8.07
Dewater	26.82	20.75
Ventilation	7.71	4.73
Cooling	8.27	5.65
Rock Handling	25.81	19.46
Services	0.67	0.53
Subtotal [MW]	91.62	68.59

18.1.4.4 Construction Power Supply

Power supply for the construction period is sourced from the existing 120 kV line. An existing 15 MVA 120/11 kV mobile transformer will be relocated to the construction area. Power for the various construction activities on site will be from a skid mounted 120/11 kV Kakula substation at 11 kV. This substation is currently located at the Konsoko West mine and will be relocated to the planned 220 kV substation yard at the Kakula mine.

18.1.4.5 Medium Voltage Electrical Distribution

The selected voltages for the Kakula 2019 PFS are as follows:

- Medium voltage systems
  - Distribution voltage: 22 kV Surface and 11 kV underground AC resistively earthed.
  - Nominal frequency: 50Hz.
- Low voltage systems:
  - Mine developing equipment operating voltage: 1000 V AC resistively earthed.
  - Motor operating voltage: 690 V AC resistively earthed.
  - MCC control voltage: 110 V AC solidly earthed.
  - Small power LV voltage: 400/230 V AC.

The power factor correction (PFC) is to be implemented at the medium voltage level to take advantage of the benefits of scale. The PFC caters for both power factor correction and harmonic filtering requirements. A distributed PFC philosophy has been applied. This solution provides greater flexibility in terms of incremental introduction as the site load increases. The MV distribution will be from the 40 MVA 220 kV/22 kV transformers and outdoor yard, and will be distributed as illustrated in Figure 18.4. Single bus switchgear will be installed In all the MV substations.

Figure 18.4 Kakula Site MV Distribution

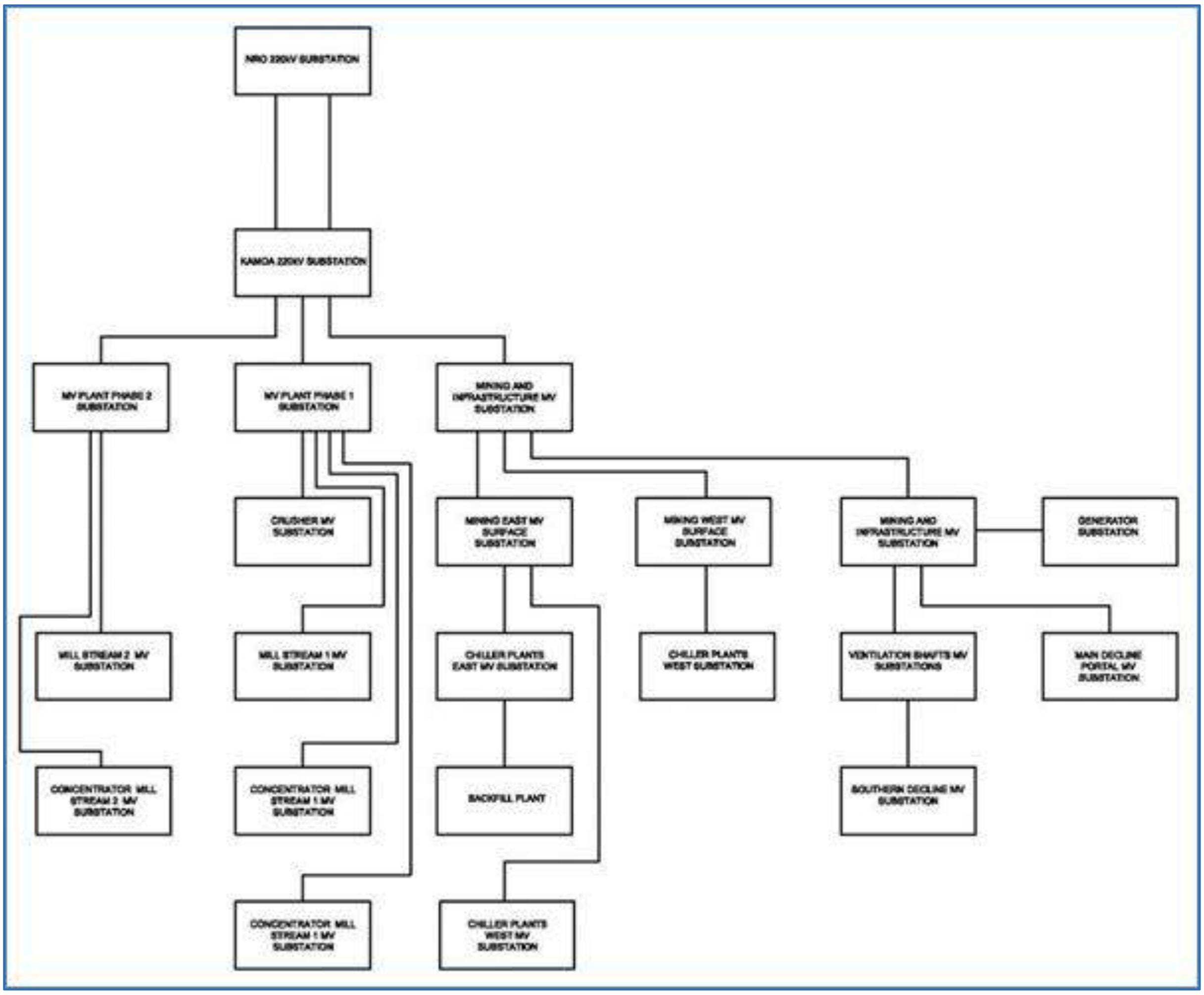


Figure by DRA, 2019.

18.1.4.6 Back-up and Standby Power

A MV Generator farm (14 x 2500 kVA MV continuous rated units) will generate backup power to the planned power system. Generated power will feed into the MV network during power outages. A Power Management System (PMS) will be commissioned to monitor all MV switchboard, synchronise the generators, switch off non - essential breakers and optimise the efficiency of the generator power plant.

18.1.4.7 Low Voltage Electrical Distribution

The MCC distribution voltage will be at 690 Volts for surface and underground infrastructure. Suitable sized transformers i.e. 2000 kVA 22/690V and 11 kV/690V Dyn11 ONAN step-down transformers will be installed. These will be connected with the neutral point resistively earthed and will be utilised to power the MCCs.

18.1.4.8 Lighting and Small Power

The non-essential lighting and small power supply in each plant area will be taken from independent sub-boards supplied from 22 kV/400V and 11 kV/400V Dyn11 ONAN transformers and mini substations. These include the numerous offices, workshops, change houses and other similar facilities. The neutral point of the 400V transformers is solidly connected to the earth. In outlying areas where there is a local MCC, small power will be taken from a 690 V/400V transformer supplied from the MCC.

Only energy-efficient forms of lighting have been utilised with respective facilities for person presence detection, and/or automated remote switching included as appropriate for further energy savings.

18.1.4.9 Control System

The control system architecture is designed around a Programmable Logic Controller (PLC), and central Supervisory Control and Data Acquisition System (SCADA). The Mining Control Room (MCR) is located on surface near the main decline area for the control of daily mining operations, on surface and underground. The Process Control Room (PCR) is located on surface at the plant area, for control of the daily plant operations.

All electrical feeds and plant status is monitored and logged and forms the basis for Power Factor Correction (PFC), Energy Management (EM) and Power Management (PM). Communication between all the relevant points will be done over optic fibre utilising overhead lines and already existing routes.

18.1.4.10 Instrumentation

The instrumentation system design, is based on equipment specifications and Control and Instrumentation (C&I) design criteria developed for the project. In general, with the exception of belt-scales and density meters that communicate via a ProfiNet (PN) fieldbus, conventional “hard wired” type instrumentation is used in the design.

Instrumentation is based on standard signal types. Instrumentation will be wired directly to weather-proof, field-mounted Remote I/O (RIO) boxes, located applicably around the plant. RIO boxes will be connected on a copper or fibre link back to the relevant control room.

18.1.5 Kakula Tailings Storage Facility

Epoch Resources (Pty) Ltd (Epoch) completed a PFS design of the Tailings Storage Facility (TSF) and associated infrastructure as part of the Kakula 2018 PFS.

The terms of reference that Epoch was responsible for include:

- The design of the TSF comprising:
  - A Tailings Dam (TD) that accommodates 53 million dry tonnes of tailings over a 26-year LOM.
  - A Return Water Sump (RWS) and Silt Trap associated with the TD.
  - The associated infrastructure for the TD (i.e. perimeter slurry deposition pipeline, stormwater diversion trenches, perimeter access road etc.).
- Estimation of the capital costs to an accuracy of ±25 percent, operating costs associated with these facilities to an accuracy of ±25 percent and closure costs to an accuracy of ±50 percent.
- Estimation of the costs over the life of the facility.

The site selection study undertaken, by Epoch, found the most favorable site as being the Kakula site.

The key design features of the TSF are as follows:

- The TD will initially be constructed as a valley containment dam with a compacted earth embankment wall, with the following features:
  - The TD will initially be constructed as a full containment facility to elevation 1445 mamsl. which will provide 2.7 Mt of capacity;
  - The five subsequent downstream construction phases will raise the containment wall by 5 m per lift;
  - The final lift, at 1470 mamsl will provide 53 Mt of capacity; and
  - The TD has a total footprint area of 370 Ha, a maximum height of 45 m and a final rate of rise of <1 m/year;
  - A curtain drain inside the Phase 1 wall, as well as a curtain drain intersecting the Phase 2 to Phase 6 walls. An outlet pipe will convey seepage water from the curtain drains, to the RWS. Inspection manholes will be provided at intersections along the outlet pipes to monitor the operation of the drains;
  - A penstock decant system with 13 intake structures and a spigot-and-socket concrete pipeline;
- An RWS with a storage capacity of approximately 1,400 m^3;
- A Silt Trap with two concrete lined chambers;
- A slurry spigot pipeline along the crest of the TD; and
- Storm water diversion (by Golder).

18.1.5.1 Project Location and Topography

The terrain is mostly grasslands with some dense pockets of trees. The general topography of the Kakula site area can be seen in Figure 18.5.

Figure 18.5 General Topography of the Preferred Kakula TSF Area

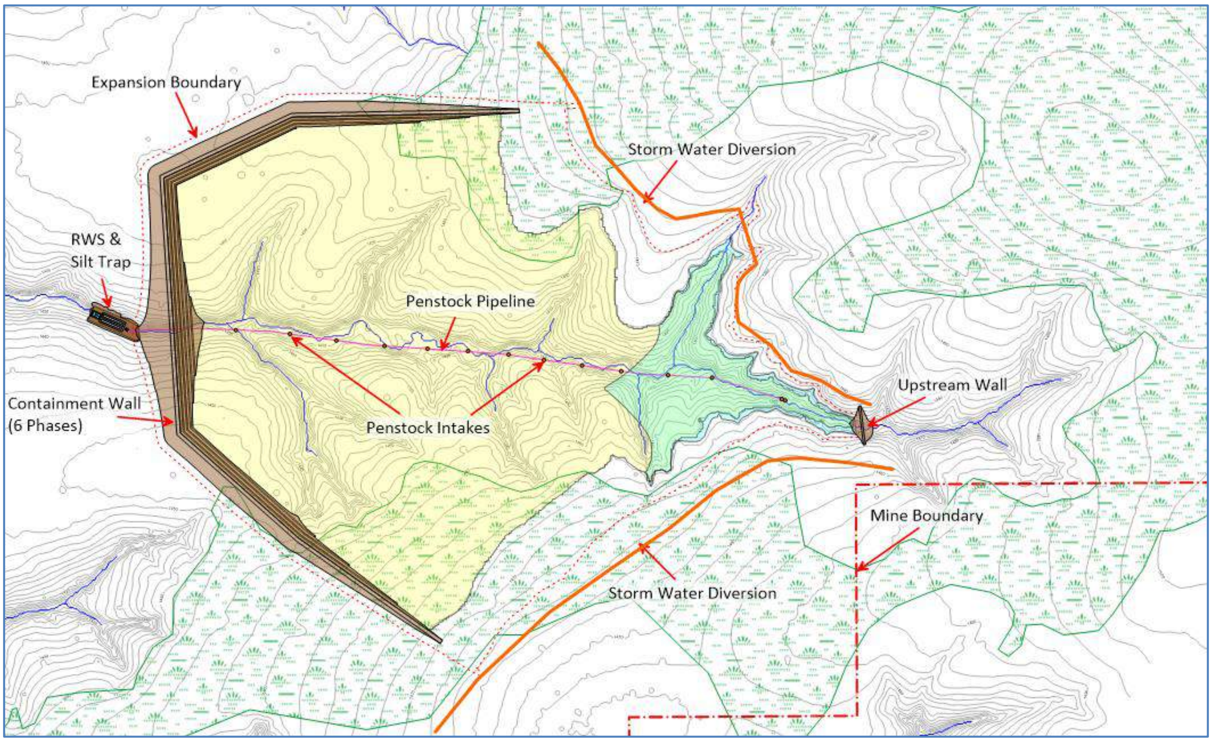


Figure by Epoch, 2018.

18.1.5.2 Design Criteria and Assumptions/Constraints

The design of the TSF was based on the design criteria shown in Table 18.2. The tailings deposition rate and PSD was determined based on the requirements for backfill into the mine, which will require 45% of the total tailings feed. Cyclone underflow (coarse fraction will report to the backfill plant and cyclone overflow will report to the TSF.

Table 18.2 Design Criteria Associated with Kakula TSF

Description	Value	Unit
Design Life of Facility	26	Years
Average Tailing Deposition Rate	2,430,000	Dry tpa
Particle SG of Tailings product	2.85	
In-situ Void Ratio	1.25	
Particle size distribution of Tailings product	100% passing 20 micron	
Average dry density of tailings	1.26	t/m^3
Site's Seismicity	0.08 g	PGA
Design Storm (24 hr, 1 in 100 year)	139	mm

18.1.5.3 Liner Requirements

Golder undertook the geochemical characterisation of the tailings and classified the tailings according to the DRC regulations. They classified the tailings as Leachable Mine Waste, due to the high concentrations of copper and iron in the Toxicity Characteristics Leaching Procedure (TCLP) test. Therefore, the DRC regulations require that the TSF be sited over soils with a permeability less than 1x10-6 cm/s or, failing this, a geomembrane should be installed.

Insufficient geotechnical information exists at present to determine if the soils adhere to the permeability requirement, therefore a HDPE liner has been assumed for the study.

18.1.5.4 TSF Site Selection

The preferred TSF site is the Kakula site. The site selection study was undertaken by Epoch as part of the preliminary economic assessment in 2017. This site was chosen for the TSF for the following reasons:

- The topography and soil properties are such that it will not require expensive measures to both contain the tailings and prevent ground water and surface water contamination.
- Due to the narrow valley, the site would result in the lowest start-up capital costs.
- The site is located within the current Kamoa tenement.
- No mineralization exists below the site and will not result in sterilization of any resources.

18.1.5.5 Design Considerations

The design of a TSF usually begins with determining what type of facility will be selected to contain the tailings. Two common types of facilities are self-raised (upstream) and full containment (downstream). Due to the fineness of the overflow tailings to report to the TSF, it will not be possible to self-raise the tailings. Therefore, a full containment facility, with a downstream construction methodology was selected.

The fineness of the Kakula tailings also has the following implications:

- Subsoil drains cannot be constructed in the tailings as they will blind and become inoperable. A full containment facility can utilise a curtain drain to reduce the phreatic surface through the wall. A curtain drain will not be in contact with the tailings and thus cannot blind.
- Return water will contain suspended solids if it is not allowed sufficient time to settle. This can be mitigated by constructing a silt trap/settling facility or maintaining a pool for longer durations on the TSF. A self- raised facility has the implication that it cannot hold large quantities of water as it will affect the stability, whereas a full containment does not.
- The beach slope of the TSF is expected to be very flat, making pool control very difficult for an operator.

18.1.5.6 Stage Capacity and Site Development Strategy

The containment wall has been phased in order to delay capital expenditure as far into the life of the facility as possible. In order to effectively phase other construction items (such as the penstock pipeline and liner), one intermediate wall which will hold the tailings back while allowing the contractor to construct and install the penstock and liner, as well as a final back wall, will be provided at the upstream side of the valley. The containment wall, penstock, liner and intermediate and back walls will have several stages which correspond to certain containment wall phases (see Figure 18.6 and Figure 18.7).

Figure 18.6 Containment Wall and Lift Phasing

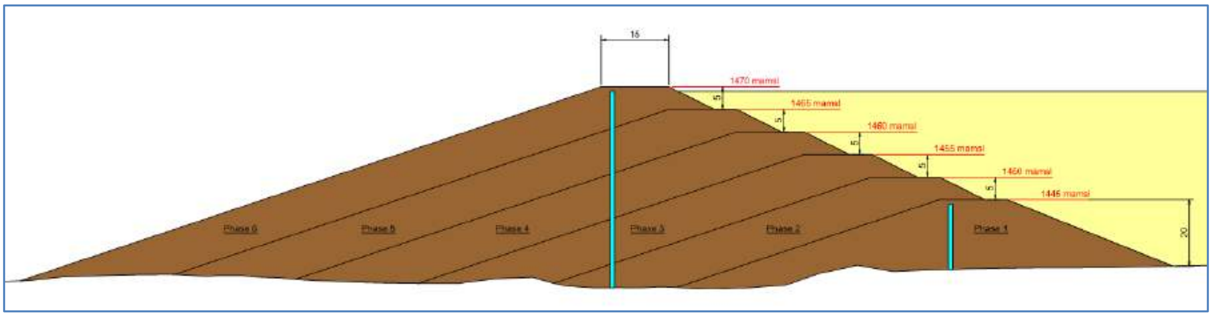
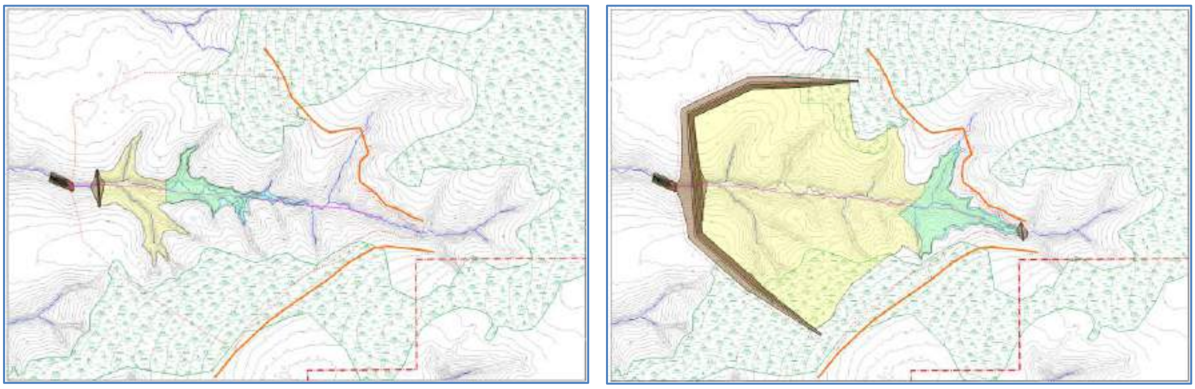


Figure by Epoch, 2018.

Figure 18.7 Containment Wall and Footprint Phasing



Phase 1 (Start-up)

Phase 6

Figure by Epoch, 2018.

18.1.5.7 TSF Construction Works

The preparatory works associated with the TD comprise the following:

- Topsoil stripping to a depth of 300 mm beneath the TD footprint;
- Termite mounds must be removed under the wall and areas to be lined with HDPE;
- A box-cut to a depth of 500 mm beneath the containment wall;
- A compacted key below the Phase 1 wall embankment;
- A compacted earth embankment with the following dimensions (Phase 1):
    - 20 m high (i.e. crest elevation of 1445 mamsl);
    - 15.0 m crest width;
    - 1V:2H internal side slope; and
    - 1V:3H external side slope.
- Two curtain drains inside the containment wall (Phase 1 wall and Phase 2 to 6), to reduce the phreatic surface through the wall. The water will be conveyed to the RWS;
- A storm water run-off trench and berm around the TSF from which run-off from the side slopes of the TD is directed away from the TD;
- Temporary storm water diversion channel upstream of the Phase 1 TD with its associated cut-to-fill berm wall;
- A buried 1050 ND Class 150D spigot-socket precast concrete penstock pipeline composed of 10 single intermediate intakes and a double final vertical 510 ND precast concrete penstock ring inlet. The penstock pipeline will be fully encased in reinforced concrete for 1400 m and half-haunched for the final 1,254 m;
- A 1500-micron liner in the basin of the TD and side slopes of the containment wall, underlain by a Geofabric;
- A 300 ND slurry spigot pipeline along the length of the TD containment wall; and
- Concrete lined: Energy dissipator, Silt Trap and RWS..

As no stability analyses have been conducted in this stage of the project, the configuration of the TD side slopes will need to be re-evaluated in the next phase of the project.

The specified size of the penstock pipeline and the slurry delivery pipeline has been based on preliminary design calculations and should be re-evaluated during the next phase of the project.

18.1.5.8 TSF Depositional and Operational Methodology

The depositional technique selected for this project will be a valley containment, hydraulically deposited, spigot facility. The containment wall will be constructed using borrow material and tailings will be deposited behind the wall and into the valley. This design is a common construction technique used in tailings storage facilities. The three principal designs are downstream, upstream and centreline structures, which designate the direction in which the embankment crest moves in relation to the starter wall at the base of the initial embankment. The Kakula TD is a downstream structure. Figure 18.8 shows a simplistic diagram of the stages of construction of a downstream raised embankment.

Tailings is usually discharged from the top of the dam crest creating a beach and a resulting supernatant pool develops as far away from the wall as possible. Where the tailings properties are suitable, natural segregation of coarse material settles closest to the spigot and the fines furthest away.

Figure 18.8 Downstream Method of Embankment Construction

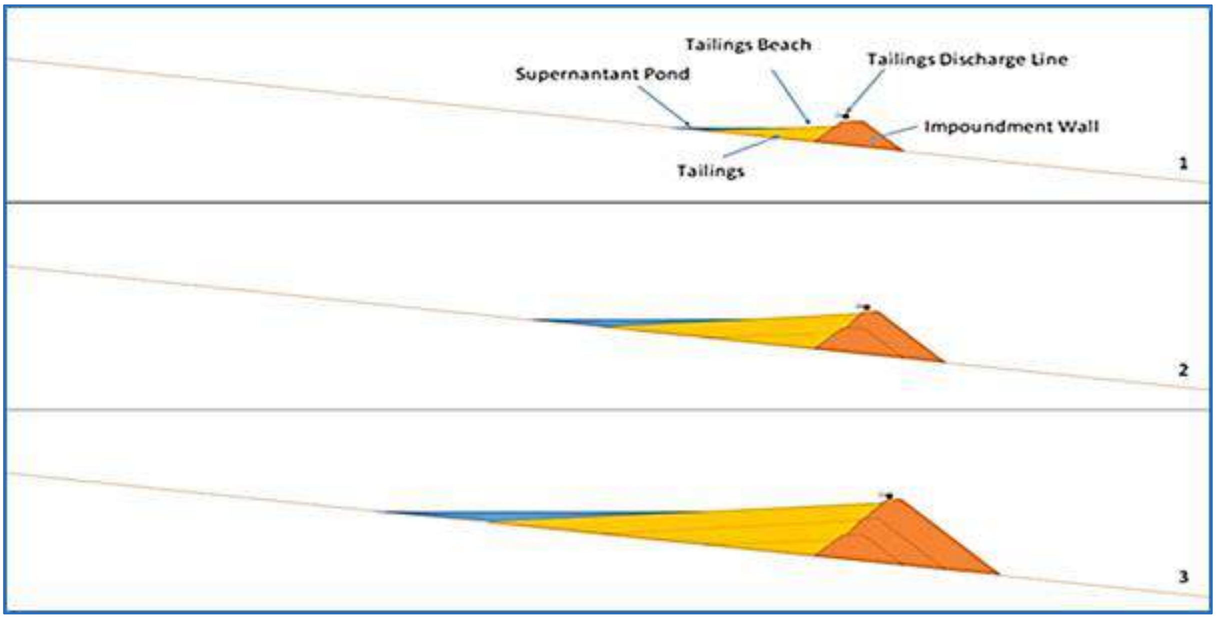


Figure by Epoch, 2017.

As the tailing is expected to be ultra-fine, more water is expected to be locked up between the tailings particles, resulting in lower densities and strength. Another consequence of ultra-fine tailings is very flat beaches, which could make pool control difficult and will require careful management by the operator.

For the selected depositional methodology, tailings are deposited into the TD basin via a spigot pipeline located on the inner crest of the perimeter wall as shown in Figure 18.9.

Figure 18.9 Multiple Spigot Discharge

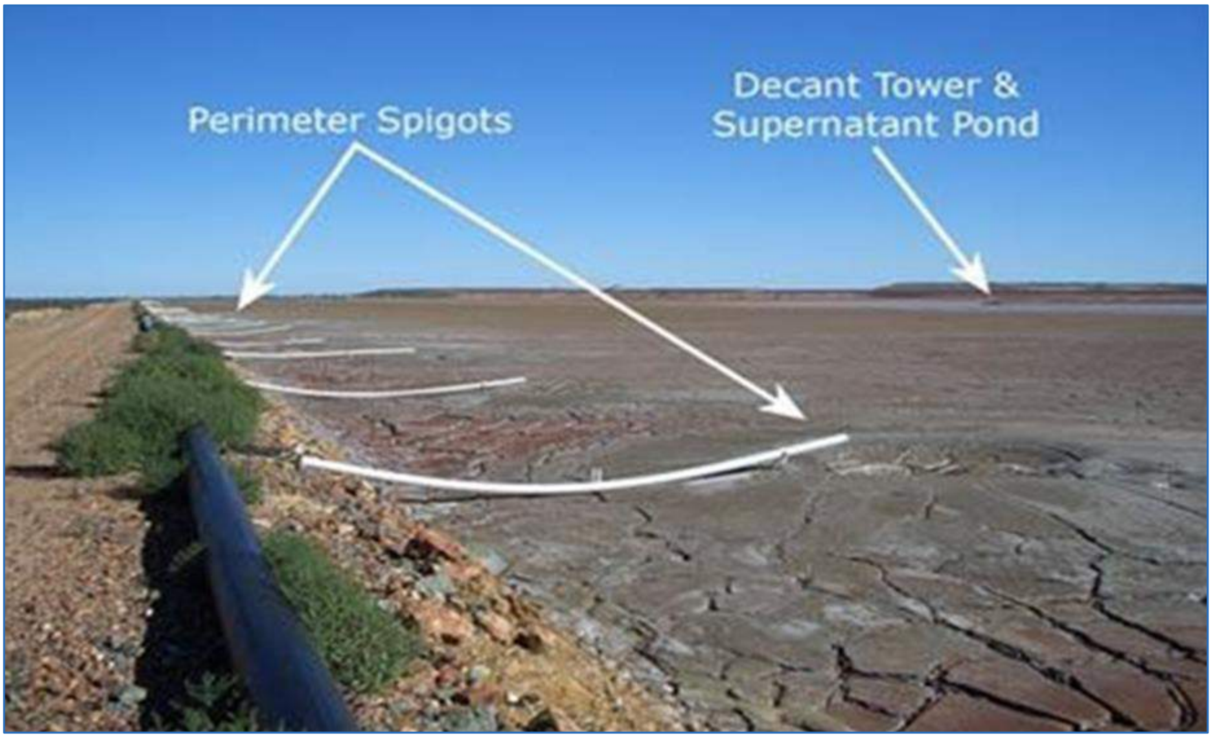


Figure by www.tailings.info, 2017.

18.1.5.9 Water Balance

A daily water balance simulation was undertaken to determine how much water can be stored on the TD, as well as the expected return water volume (as a percentage of the water in the slurry reporting to the TSF). The results of the simulation are shown in Figure 18.10.

It was determined that during the wet months, sufficient capacity may be available on the TD, to limit the return water to 100%, while slowly decanting/evaporating the water during the dry months.

Figure 18.10 Water Balance Results

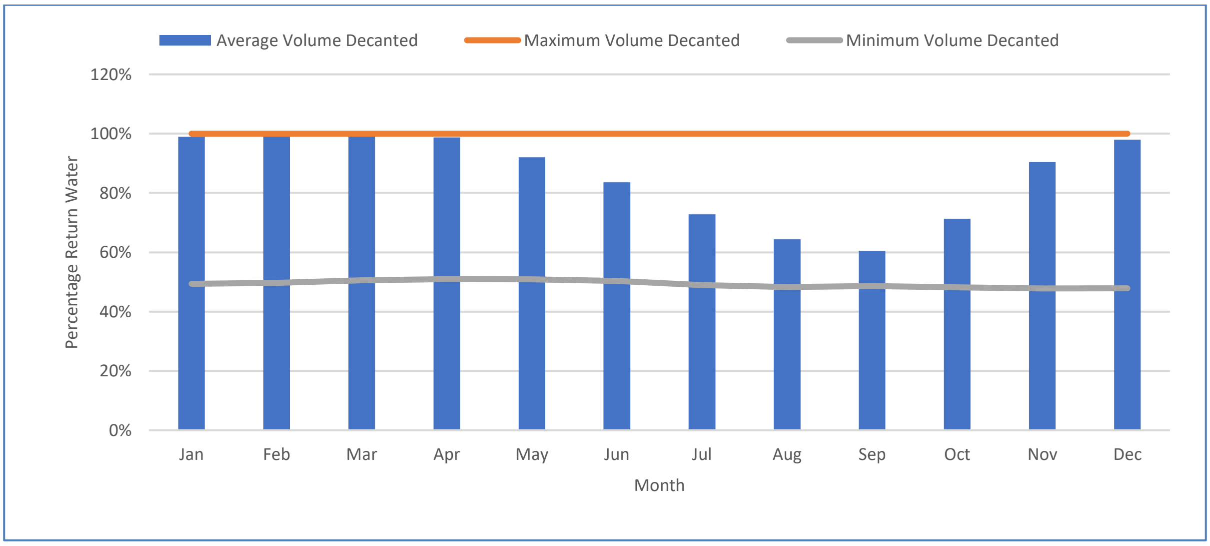


Figure by Epoch, 2018.

18.1.5.10 Closure Activities at Cessation of Operations

At the cessation of operation of the TSF, the focus will be on the cover and vegetation of the top surface of the facility, the decommissioning of facilities associated with the TSF and the construction of storm water and erosion control measures as required.

The duration of the final closure process may be affected by the length of time required for the basin of the facility to dry sufficiently to enable the placement of cover material in preparation for the vegetation establishment.

18.1.5.11 Risks

The possible project risks associated with the current TSF design are as follows:

- Kakula is situated in a seismically active area. No stability analyses have been undertaken for the TSF to confirm that the current TSF geometry will withstand a seismic event.
- A suitable borrow pit has not yet been identified for use in the containment wall.

18.1.5.12 TSF Recommendations

For the Definitive Feasibility Study stage of the project, it is recommended that the following be included:

- Validation of the suitability of the TSF site by the Environmental and Social Impact Assessment and other specialist studies;
- A geotechnical investigation of the TSF site in order to confirm the type, extent and characteristics of the in-situ materials as well as available construction materials;
- The identification of borrow pit areas within the project site are required for sourcing of material for the construction of the TSF starter walls;
- A seepage analysis and slope stability study be undertaken to confirm the seepage regimes through the TD as well as to confirm the TSF stability. The results of these analyses could impact greatly on the geometry of the TD walls and ultimate height of the facility;
- Confirmation of the physical characteristics of the tailings product based on laboratory testing of a representative sample;
- Possible further optimisation of the TSF preparatory works in terms of layout, footprint extent, etc. including any changes to the mine plan;
- A detailed water balance should be undertaken to ensure that the water storage capacity on the TD and the decant volumes balance, to accurately predict the water volumes reporting to the process plant or water treatment plant. This should include any changes to footprint areas, rainfall data, slurry water/solids ratio, etc.;
- Review the construction rates with a contractor to price the facility with representative rates; and
- Compilation of a detailed schedule of quantities describing the proposed preparatory works and the pricing of the schedules to a greater level of accuracy.

18.1.6 Site Waste Management

The waste management system was designed to cater for 3000 people at the peak. Over the project life of 30 years, the general waste generated by the mine as well as domestic waste generated by the mine personnel is to be disposed at a landfill.

A full description of the waste management and requirement is referred to in the Kamoa – Landfill Re-Design report done by Golder.

18.1.7 Roads and Earthworks

18.1.7.1 Terraces

Golder Associates Africa (Pty) Ltd (Golder) was appointed by Kamoa Copper SA (Kamoa) to provide geotechnical engineering consulting services for the Kakula 2019 PFS. Observations from geotechnical fieldwork, as well as analysis and interpretation of the laboratory testing results on the soils sampled, were used for the various classifications. Geotechnical conditions encountered at the site were classified and foundation requirements were outlined for planning of surface mine infrastructure purposes.

Due to the geotechnical conditions in the mine area and the different bearing capacity requirements, it necessitates splitting the terraces into three terrace groups i.e. low, medium and high specification.

Low specification terraces will be used in laydown areas and temporary terraces required for the construction phase of the project, with a bearing capacity of 50 kPa to 100 kPa.

Medium specification terraces cater for typical buildings and stores, where the design requires a bearing capacity of 150 kPa. The following areas are identified as areas requiring medium specification terracing:

- Workshops;
- Offices;
- Stores;
- Change house;
- Gate houses, and;
- Waste facilities.

High specification terraces cater for large and high structures or vibrating structures where the design requires a bearing capacity of 250 kPa. The following areas are identified as requiring high specification terracing:

- All concrete stockpile structures.
- Crushing and screening structures.
- The mill, flotation, thickeners and filter building in the process plant area.

18.1.7.2 Roads and Parking

The main access road is currently being constructed as part of the Kakula project to allow for future mining activities required for the Kamoa-Kakula Copper Project. This road gives access from Kolwezi to Kakula mine and is divided into two sections:

- Section 1: Section from Kolwezi Mines turnoff to Kansoko.
- Section 2: Kansoko Mine to Kakula Mine.

The internal road design philosophy is that delivery vehicles, LDV's and concentrator trucks will remain on separate roads to the required delivery points, working and parking areas. With internal roads reserved for the delivery of equipment from stores to the applicable work area. The internal roads and parking take account of the traffic flow inside the mine area. Security gates separate areas in order to control access, without reducing serviceability and production.

Different types of surface finish and layer works have been designed for differing types of application and road uses within the mine area. The following have been used as part of the mine design and layout:

- Asphalt road layer works used for the main entrance access road,
- Paving road layer works where heavy equipment and trucks are turning, and
- Gravel road layer works used mainly in the outer portions of the concentrator plant and site infrastructure for maintenance access.

18.1.7.3 Storm Water Management

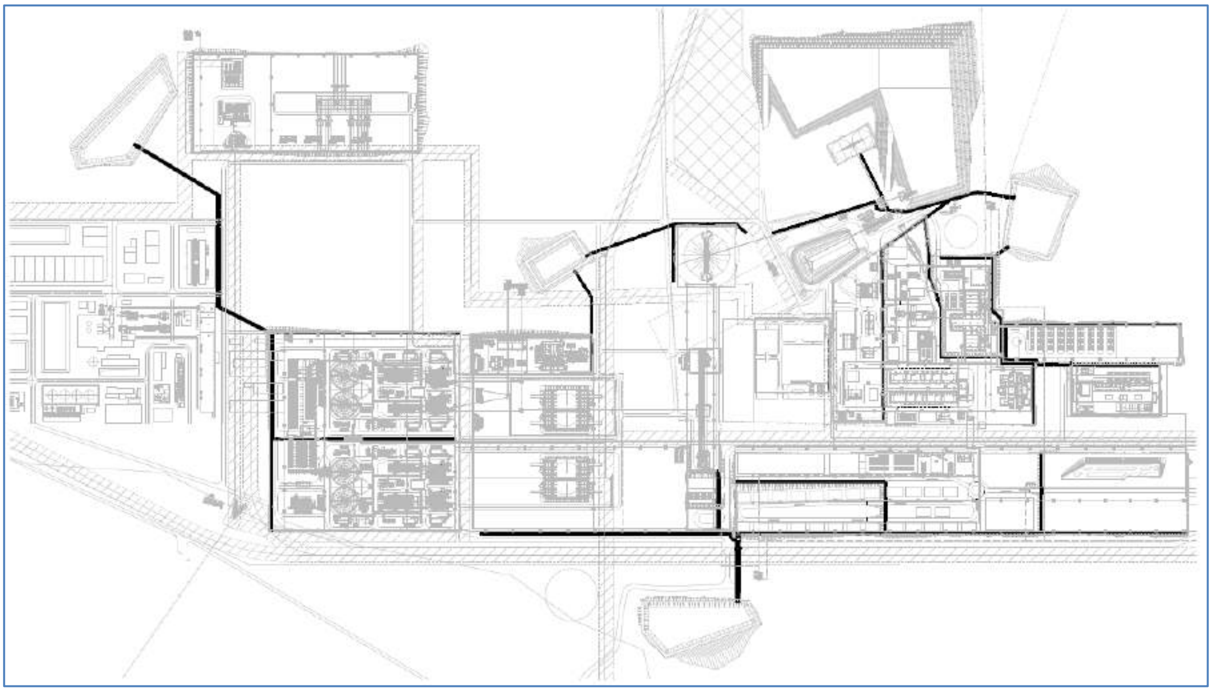
Golder Associates Africa (GAA) modelled the run-off and made recommendations, based on the following:

- A one-in-hundred-year flood line is applied and all structures on the mine will be protected against this. A one-hundred-year flood line is a line drawn on a contour plan showing the edge of the water level of a river during flood condition.
- A one-in-fifty-year-flood/storm water event (1:50) was used to calculate the storm water run-off and peak flow, to size the required storm water infrastructure and design thereof. This is a flood event that has a 2% probability of occurring in any given year.
- Freeboard of a minimum 0.8 metres has been applied. Freeboard with respect to water storage dams can be defined as the distance between the full supply level (spillway crest level), and the lowest point on the dam wall crest.

The storm water management system consists of storm water run-off drains, storm water dams, and a discharge drain. The storm water run-off drains are a network of drains running through the mining area collecting all run-off water and directing it towards the storm water Pond 3. These drains vary in size and all are concrete lined.

Figure 18.11 shows the layout of the primary collection drains. Additional secondary drains have been positioned to channel all run-off water.

Figure 18.11 Storm Water Run-off Drains



Discharging of the collected clean water into the nearest river, will be via a discharge drain, designed to minimise potential flooding of surroundings.

Dirty water collected in storm water dams will discharge for events over and above 1:50 to the nearest watercourse.

18.1.7.4 Water Storage Facilities

Different types of dams have been allowed for in accordance with their varied requirements. Table 18.3 indicates the water storage facilities allowed for, and their associated capacities.

Table 18.3 Water Storage Facilities and Capacities

Water Storage Facility	Material of Construction	Capacity
Storm Water Pond 1	Earth dam	32 ML
Storm Water Pond 2	Earth dam	47 ML
Storm Water Pond 3	Earth dam	17 ML
Storm Water Pond 4	Earth dam	11 ML
Tailings Dam	Earth dam	1,000,000 m^3
Return Water Sump	Earth dam	1,400 m^3
Raw Water Dam	HDPE lined earth dam	15 ML
Mine Service Water Tank	Mild steel	5,000 m^3
Process Water Tank	Concrete	5,000 m^3
Concentrator Raw Water Tank	Concrete	1,000 m^3

18.1.7.5 Material Storage Stockpiles

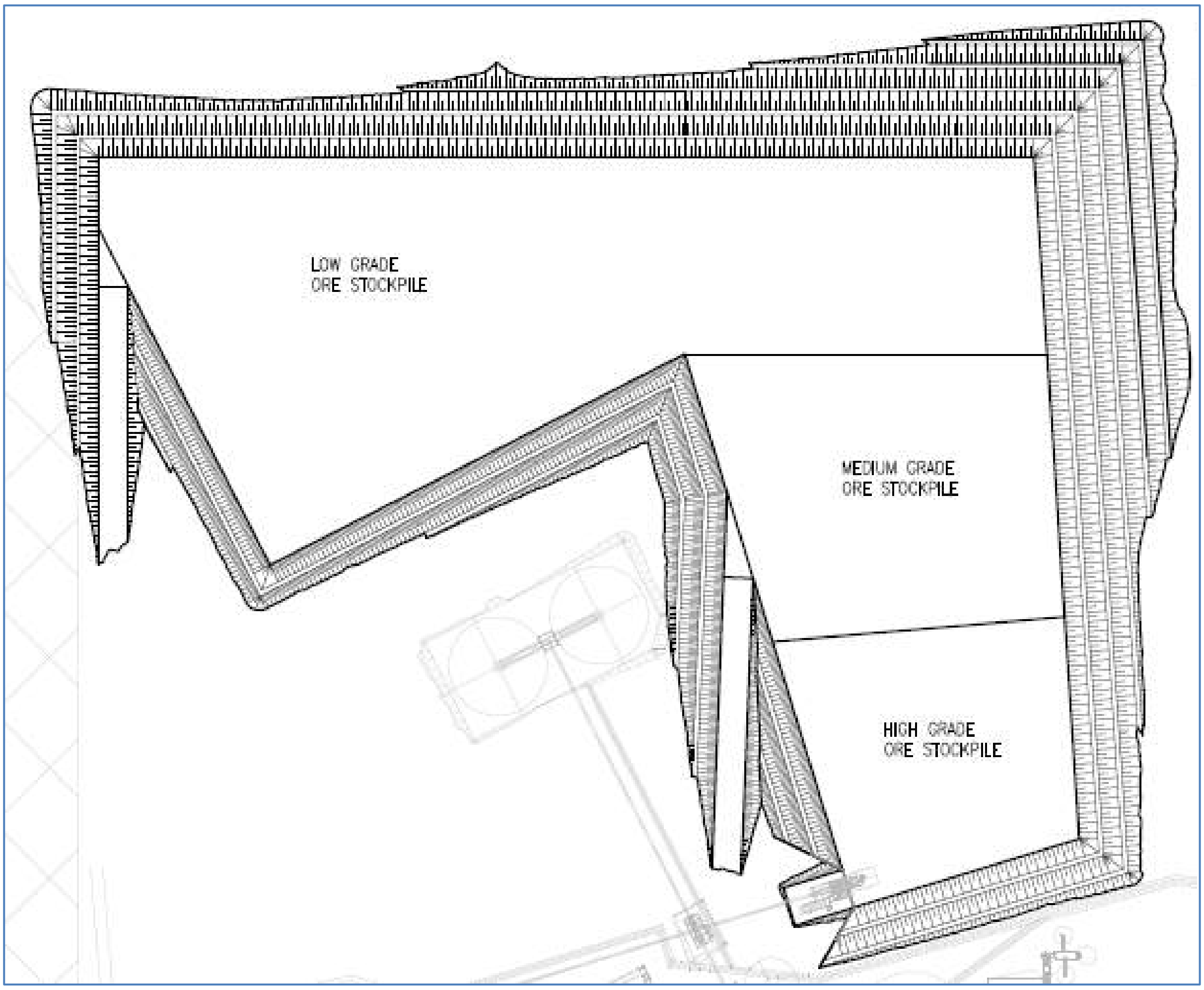
Both ore and waste rock stockpiling encompasses the stockpiling of all commodities generated during both construction, and operational phases. These commodities include:

- Low-grade (LG) ore,
- High-grade (HG) ore,
- Waste rock uncrushed and crushed, and
- Subsoils (soft and hard).

Materials deposited onto temporary stockpiles are spread further by means of mobile earthmoving equipment (trucks, loaders and dozers) to build individual type stockpiles. Stockpiles have been designed with the required footing and drainage. The footprint of each stockpile was modelled to optimise the required capacity.

A further requirement for waste rock is to re-use this as earthworks backfill material. To cater for this, waste rock has to be crushed and screened to suitable sized materials. A mobile crushing and screening plant has been allowed to facilitate this. Refer to Figure 18.12 below for and illustration of the bulk ore stockpiles.

Figure 18.12 Material Storage Stockpile



18.1.7.6 Earthworks Commodity Flow Philosophy and Strategy

On account of the various earthworks commodities required during the construction phase, a basic commodity flow model was developed for the project, as this directly impacts on the overall capital estimate. The objective of this model is to indicate the nett quantities of material/commodities requiring to be stockpiled, whether temporarily or permanently. It further indicates commodity requirements, and if importing of commodities is required onto the site.

18.1.7.7 Buried Services

All buried services are designed according to SANS 1200. These include:

- Earthworks, i.e. trenching (SANS 1200D).
- Bedding for pipes (SANS 1200LB).
- Piping, valves and valve chambers, anchor/thrust blocks and manholes (SANS 1200L).
- Concrete and miscellaneous metal work (SANS 1200G).

18.1.7.8 Sewer Reticulation

The domestic sewage requirements for the mine area is designed to enable collection from various facility points, and outed through a 100 to 250 nominal bore (NB), depending on the flow, polypropylene gravity fed buried pipe system, to a concrete-lined sump at the sewer plant. The design capacity for this sewer treatment plant is to cater for 3000 people.

18.1.7.9 Weighbridge

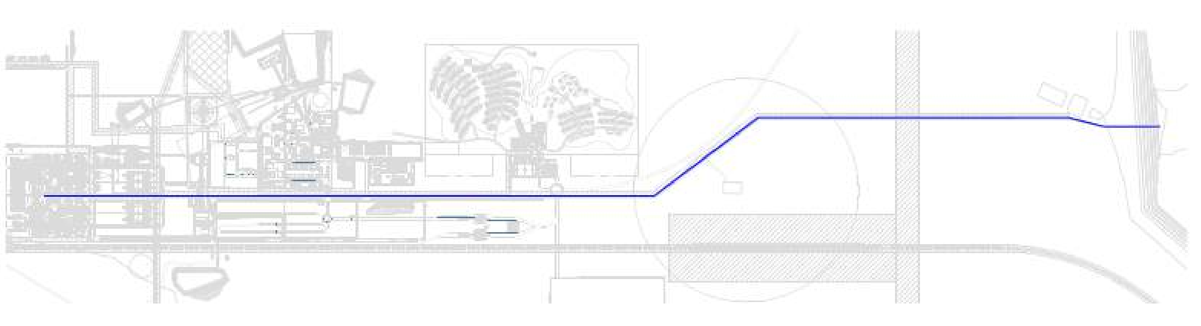
Three weighbridges have been included in the design. Two are allocated after the main gate before the bonded yard. One is in the plant area specifically to measure concentrate and materials.

All weighbridges are positioned enabling vehicles entering or leaving the area to be weighed without disrupting traffic flow.

18.1.7.10 TSF Pipeline Servitude

The tailings line will be phased in line with construction of each of the 3 Mtpa concentrator’s. The HDPE tailings pipelines will be routed within the tailings servitude and will be surface run and unsupported and unlined. A 10 m servitude will be bush cleared for the pipelines. The tailings line servitude is shown in Figure 18.13 below.

Figure 18.13 Tailings Pipeline Servitude



18.1.8 Security and Access Control

Security of the site is regulated by way of access control, closed-circuit television (CCTV) and a security alarm. The access control system is software-based which works as a dual tag system, meaning both fingerprints and a card reader will be required to grant a personnel entry or exit to and from the site. The CCTV system and the access control system will be integrated at all main entry and exit points. The security alarm system is easy to service and maintain. The Mining Lease Area is fenced off along the outside of the perimeter berm and various access control buildings have been allowed within the mine area. A summary of the security and access control infrastructure is illustrated in Table 18.4

Table 18.4 Security And Access Control Infrastructure – Overview per Area

Mining Area	Concentrator Plant	Kakula Camp	Weighbridges	Security Main Control room
CCTV Control room - guardhouse (office equipment, computers, screens CCTV etc)			CCTV (amount to be determined)	CCTV Control room – Mag lock doors fitted with biometric zone control integrated (office equipment, computers, camera, screens CCTV etc)
Electronic/motorised boom gates fitted with RFID tags / biometric readers, gooseneck pedestals for car and truck height reader points fitted with RFID tags / card, biometric readers and cameras / back up batteries			RFID tag system	NVR channel 24 x (amount to be determined) Recording server room
4 arm Turnstiles (man trapped) power supplied and battery back up			Robot access control system at weighbridge	Security Alarm system
Single Bio-metric readers / pass back control / Zone control			Gooseneck pedestals for truck height reader points fitted with RFID tags / card, biometric readers and cameras / back up batteries	Strike locks and bio-metric readers integrated
Alcohol blow machines / breathalysers connected to turnstiles				
Day/night Cameras vandal resistant inside facilities and at high risk areas				
Fencing around area – Monitored - Early Warning Detecting alarm system (Illumination) Clear view fencing with razor coil attached				
Search rooms (CCTV and audio)				
UPS			UPS	
Office alarm system connected with motion detection CCTV				
Visitors Security reception room – computerised access control system			Computerised access control system	
	Lab complex – biometric access zone control, mag lock doors	Kitchen Access Control- cameras, bio-metric readers and turnstile / pass back control /zone control / meal counts (complete access control interface)	Biometric access control system / Driver / truck identification	Complete TAS solution (TIME / ACCESS / SURVEILANCE integrated system)
				Badge making and printing system
				Fingerprint enrolment system, cameras, stand and facility

18.1.9 Logistics

There are several alternative logistics routes available for the transportation of copper concentrates from Kamoa-Kakula to regional smelters and ports.

Currently the North South corridor between southern DRC and Durban or Richards Bay in South Africa is viewed as the most attractive and reliable export corridor. Product would be transported by truck to Ndola in Zambia and then loaded on to trains for onward transport to the ports of Durban or Richards Bay in South Africa. This forms the base case for the PFS.

In future, there is the possibility to use the existing 2,000 km rail line between Kolwezi and the Angolan port of Lobito. This line has been re-built for 1,600 km between Lobito and the Angolan-DRC border at the town of Dilolo and can handle a capacity of 20 Mtpa. The 400 km on the DRC side of the border, from Dilolo to Kolwezi, is in a poor condition and needs major repair and upgrades. As soon as this section has been sufficiently rehabilitated and put into operation, Kamoa would need to construct a private 20 km rail spur linking the mine to the main line and product will be railed directly from the mine to Lobito for export.

A number of alternate export corridors will remain available to Kamoa and could be used if necessary. Apart from the North South corridor to Durban and the Lobito/Benguela corridor to the West, the Tazara corridor to Dar es Salaam in Tanzania and the option of exporting some volume through Walvis Bay in Namibia also exist.

Concentrate will be bagged at the mine and road hauled to the closest facility in Northern Zambia where freight can be transferred from road to rail. A number of road hauliers are active on this route. It has been assumed that a new intermodal (road to rail) facility will be available in Chingola, 45 km by road from the DRC/Zambia border at Kasumbalesa. Zambia Rail (ZRL) and a number of private logistics companies are considering developing more rail linked facilities further north of Kitwe, which is currently the northern most and closest rail linked facility to the border with the DRC. Bagged concentrate will then be packed into 20 ft containers at the port.

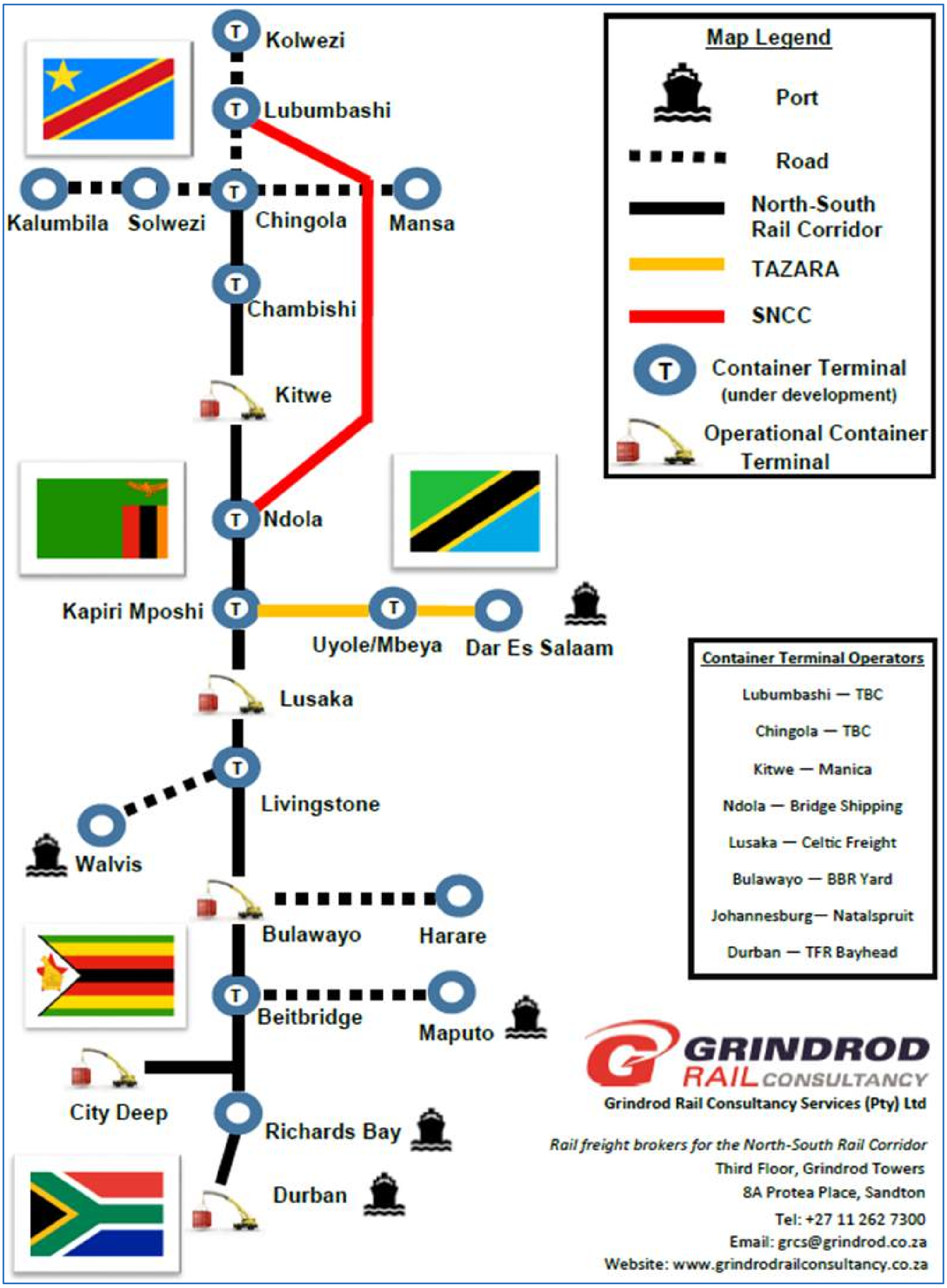
The western rail corridor to Lobito and the North–South corridor through to Zambia is shown geographically in Figure 18.14. The North–South Corridor is shown in diagrammatically in Figure 18.15.

The use of an operational line between Kolwezi and Lobito port is not exclusively dependent on the rehabilitation of the rail infrastructure. It needs joint agreement from both countries' respective governments, in addition to completing an institutional framework that should govern these operations. It also requires the DRC national rail authority (SNCC) to award a private concession to upgrade and operate the rail.

Figure 18.14 Kamoa to Lobito Rail System

Figure by Grindrod, 2017.

Figure 18.15 DRC to South Africa North–South Rail Corridor



Container Terminal Operators

Lubumbashi — TBC
Chingola — TBC
Kitwe — Manica
Ndola — Bridge Shipping
Lusaka — Celtic Freight
Bulawayo — BBR Yard
Johannesburg — Natalspruit
Durban — TFR Bayhead

Grindrod Rail Consultancy Services (Pty) Ltd

Rail freight brokers for the North-South Rail Corridor
Third Floor, Grindrod Towers
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Tel: +27 11 262 7300
Email: grcs@grindrod.co.za
Website: www.grindrodrailconsultancy.co.za

Figure by Grindrod, 2015.

18.1.10 Airports

Lubumbashi International Airport in DRC has an elevation of 1,197 m above mean sea level. It has one runway designated 07/25 with an asphalt surface measuring 3,203 m by 50 m. This airport is regularly serviced by the following airlines: South African Airways (operated by South African Express), ITAB (DRC domestic airline), Kenya Airways, Ethiopian Airlines, Congo Express, and a number of smaller airlines and private charters.

The Kolwezi airport is located about 6 km south of Kolwezi. The airport has an elevation of 1,526 m above mean sea level. It has one runway designated 11/29 with an asphalt surface measuring 1,750 m by 30 m. This airport is largely serviced by Air Fast, providing 4 flights a day between Lubumbashi and Kolwezi. There are plans by the Lualaba Provincial Government to upgrade Kolwezi airport to an international airport and to lengthen the runway to be suitable to receive aircraft from Europe and South Africa. It is currently possible to make special arrangements for charter flights to fly directly from Johannesburg to Kolwezi. When the Kamoa passenger numbers increase sufficiently during construction it is planned for Kamoa to operate such a service 2–3 times a week.

Kamoa is currently in the process of upgrading some office and waiting room facilities at Kolwezi airport and these premises will be rented from the Airport Authority for Kamoa’s exclusive use. The airports will be utilised to transport people, goods and material to the project site during construction and operations phases.

18.1.11 Consumables and Services

18.1.11.1 Fuel

Transport fuel and fueling infrastructure is available along all of the required routes to Kolwezi, albeit fuel quality and standards between countries are likely to vary. 50 ppm Sulphur fuel is readily available in the region. On site, it is planned for 2 or 3 fuel depots and filling stations to be owned by Kamoa and operated on a consignment basis by fuel suppliers. It is planned to have two weeks of fuel storage on site to enable uninterrupted operation during periods of delivery delays.

18.1.11.2 Maintenance

Workshops facilities will be constructed at Kakula for activities including vehicle repairs and major overhauls, a boiler making shop and a machine shop. Kakula needs to be relatively self-sufficient in terms of workshop facilities. However, there are some major OEM workshop facilities operated or being constructed by, as well as various smaller general workshops in Lubumbashi, Likasi, and Kolwezi.

18.1.11.3 Inbound Project Logistics

The provision of logistics services should be structured in a way that will best negate the risk associated with transport and freight forwarding for the project. To achieve this, a primary freight forwarding contractor should be appointed for the international component of the route. A secondary partner should be considered, to assist with supply from South Africa and other over flow requirements, if required. A local DRC customs clearing/broker partnership should also be established. It should further be ensured that the applicable protocols are implemented to allow goods to move on a duty-free basis between countries of supply and/or transit. Central warehousing facilities should be set up, to consolidate transport loads and to ensure that bonds are not retained on shipping containers. A bonded area on site has been allowed for.

There are no major road restrictions in terms of load sizes and masses for transporting equipment to site. The 2 bridges between Lubumbashi and Kolwezi that were a restriction in the past have been upgraded to carry abnormal loads.

Currently freight from South Africa to Kamoa takes about 3 weeks, including customs clearing. During construction, it will be critical to implement an efficient logistics process flow, expediting and tracking system to avoid construction delays.

18.1.12 Water Use, Treatment and Discharge Requirements

18.1.12.1 Bulk Water Requirements

There is a water shortfall during the first two years of construction, after which the project has excess water that needs to be discharged to the environment. Discharge to the environment of treated water occurs from either the Water Treatment Plant or the Sewerage Treatment Plant.

During the construction period, water is required for actual construction, dust suppression and potable water. Water requirements during the construction period peaks during 2021.

Raw water is used in the Concentrator Plant for tails and concentrate thickening, flocculent dilution, reagent mixing, dust suppression, gland service water, reagent mixing and high-grade cleaner sprays. Process water is used in producing the slurry prior to the flotation circuit as well as for flushing and hosing purposes. Potable water is required in the plant area for the ring main as well as for fire water.

The Backfill plant requires raw water for use as gland service water, reagent mixing and to flush the backfill pipes into the underground workings.

The mining area requires raw water for dust suppression on surface as well as surface reticulation to the light (LDV) and heavy (HDV) vehicle wash bays. Underground service water reticulation is required for drilling, bolting, and hosing purposes. A surface service water ring main serves the wash bays, workshops and vent shafts. Potable water is required for fire protection of the surface and underground mining infrastructure, gland service water for dewatering pump stations and reticulation as drinking water. Underground dust suppression also requires potable water.

Potable water is required throughout the general infrastructure area for drinking and fire water purposes.

18.1.12.2 Bulk Water Source

Raw water for the Concentrator plant is stored in the raw water tank. The main external source of raw water is the raw water dam. Feed to the raw water dam is via a borehole field and rainfall. It is estimated that ten (10) boreholes will be required to alleviate the water shortfall during the construction period.

Process water for the Concentrator Plant is stored in the process water tank. The main sources feeding the process water tank are the four (4) storm water dams and the return water dam that routes return water from the tailings dam. Run-off and rainfall feed the storm water dams. The tailings dam receives water from the tailings being pumped from the Concentrator Plant, as well as run-off and rainfall.

Raw water from the raw water dam feeds the potable water treatment plant which in turn feeds potable water tanks throughout the Concentrator Plant, Mining and General Infrastructure areas. Fire water tanks in each area are fed form the respective potable water tanks.

Service water for the mining area is stored in the mine service water tank. The main sources of service water that feed the service water tank are the raw water dam, surface and underground settling systems.

DRA has allowed for all the piping and pumping systems required for routing bulk water throughout the Mining, Concentrator Plant and Infrastructure areas.

18.1.12.3 Surface, Groundwater, and Mining Water Quality

Surface water is of ideal chemical quality to use for all purposes from drinking to construction provided that the bacteriological aspects is investigated and controlled if required.

Groundwater is of good quality for use in drinking, construction and raw water supply for operations with the following notes:

- Kalahari water – excellent quality but need PVC casing to avoid Fe bacteria colonies formation;
- Upper diamictite – excellent quality for all uses;
- Lower diamictite (mine zone) – elevated pH thus hard water is less ideally suited for construction and drinking but currently does not require any treatment;
- Sandstone – excellent quality but presence of slightly elevated aluminium and arsenic. The levels of these two trace metals are to be monitored over time. This water is ideally suited for construction purposes.

With regards to the mining water, It is considered that the lower diamictite water will mostly be encountered during mining but that some mixing with the sandstone aquifer (footwall) water will occur. Higher pH water however is to be expected from mine dewatering but no treatment before discharge currently is envisaged.

18.1.12.4 Water Treatment Plant

The quality of water will be monitored and if necessary managed by a water treatment plant. The PWTP package water treatment plant constitutes a large range of containerised and skid mounted systems for Surface Water and Potable Water Production in compliance with SANS 241 and WHO requirements. The process utilises coagulation, flocculation, lamella clarification, sand filtration and disinfection using chlorination. This process aids in the reduction of TSS, turbidity and organics. Containerised water treatment plants are specifically designed to supply treated water to isolated communities or locations. Further options are available to reduce COD, colour and taste as well as the addition of standby equipment and monitoring capabilities.

18.1.13 Buildings

The following building types have been included in the Kakula infrastructure design:

- Architectural buildings are typically constructed of brick, complete with steel windows and wooden/steel doors. The roof is constructed of steel and timber roof trusses, with inverted box rib (IBR) roof sheeting. Included in the buildings are all small power, lighting and furniture. These buildings were sized using estimated man power on the mine.
- Modular buildings are prefabricated buildings that consist of repeated sections erected on site.
- Workshops are typically a sheeted steel structure building with civil bases, plinths and a surface bed. These buildings also have filled in brick work on the sides, brick offices and small stores on the inside to accommodate personnel working in this building. Roller shutter doors, standard doors and windows are included, together with small power, lighting, general tools/equipment, furniture and where needed, an overhead crane.
- The workshops and stores where sized using historical data and supplying three months of storage space for consumables.
- The change house building is a sheeted steel structure with civil bases, plinths and surface bed filled-in brick work with all the necessary sanitary facilities, seating benches, light fittings, extractors, shelves and geyser/boilers.
- The building has been designed to encompass other facilities such as the lamp room, boot wash, laundry room, store rooms and offices. In addition, the design provides a logical, sequential flow for employees who are either starting or ending a shift, while ensuring safety of the employees.
- All MV substation buildings are structural steel with IBR roof sheeting, elevated concrete slab and filled-in brick work.
- All LV substation buildings are shipped containerized and supplied to site. Only civil bases will be casted on site with steel platforms for access.

18.1.13.1 Mining Area Buildings

Below is a list of all the mining area buildings allowed for:

- Change house complex (5,000 m^2).
- Engineering workshop (3,600 m^2).
- Mine store (1,763 m^2).
- Heavy vehicles workshop (1,439 m^2).
- Light vehicle workshop (1,079 m^2).
- Lamp room(1,000 m^2).
- Sub component store (720 m^2).
- Tyre change area (529 m^2).
- Items for return to OEM'S (432 m^2).
- Permanent explosives transfer shed (380 m^2).
- Mine control room (378 m^2).
- Gate house for change house (282 m^2).
- Gate House for mining area (282 m^2).
- LV substations (10 x 210 m^2 each).
- Mine offices (200 m^2).
- Heavy vehicle wash bay (175 m^2).
- Light duty vehicle wash bay (175 m^2).
- Emergency control room (110 m^2).
- Oil and lubrication store (100 m^2).
- Canteen (100 m^2).
- Office material transfer area (100 m^2).
- Gas store (100 m^2).
- Medical room (87 m^2).
- Mine rescue room (87 m^2).
- Outside yard area (50 m^2).
- Surface mini substation (4 x 45 m^2 each).
- Ablutions (3 x 35 m^2 each).

18.1.13.2 Concentrator Area Buildings

Below is a list of all the concentrator area buildings allowed for:

- Milling area open storage yard (1,500 m^2).
- Concentrator control room (975 m^2).
- Plant workshop (937 m^2).
- Plant office (363 m^2).
- Plant change house (231 m^2).
- LV Substations (7 x 210 m^2).
- Lubrication room at surface crushing building (170 m^2).
- Plant entrance gatehouse (123 m^2).
- Compressor house (100 m^2).
- Blower house (100 m^2).
- Lubrication rooms at milling area (4 x 70 m^2).
- Concentrate loading weighbridge office(60 m^2).
- Ablutions buildings (3 x 50 m^2).

18.1.13.3 General Infrastructure Buildings

Below is a list of all the general infrastructure area buildings allowed for:

- Main stores (4,223 m^2).
- Engagement and training centre (3,252 m^2).
- Main office (3,200 m^2).
- Laboratory complex (2,500 m^2).
- Bulk consumable storage for reagents (2,500 m^2).
- General meeting area (1,074 m^2).
- Sub components store (600 m^2).
- Core yard (600 m^2).
- Cement store (600 m^2).
- Truck stop shops (500 m^2).
- EMS complex (452 m^2).
- Ablutions at truck stop (450 m^2).
- Gate House at main entrance (300 m^2).
- Bonded Yard (300 m^2).
- MV substations for mining & infrastructure(18 x 210 m^2).
- Lubricant store (200 m^2).
- Canteen (150 m^2).
- Ablutions at main gate (100 m^2).
- Gas store (100 m^2).
- Chemical store (100 m^2).
- Hazmat store (100 m^2).
- Offices - Main Weighbridge Office (60 m^2).
- Gate house at bonded yard (60 m^2).
- Ablution at bonded yard (60 m^2).
- Ablution at storage yard (60 m^2).
- Paint store (60 m^2).
- Gate house at truck stop (38 m^2).
- Gate House at magazine (38 m^2).
- Gate House at waste facilities(38 m^2).
- Gate House at mock-up entrance (38 m^2).

18.1.14 Fire Protection System

The fire detection and suppression guidelines used in the PFS were generated from a combination of the following:

- Minimum requirements as set out by the NFPA codes of practice,
- Acceptable methods adopted on previous projects,
- Guidelines followed on a previous project, and
- Industry standards as well as the adoption of modernised equipment.

Fire water will be supplied to a number of ring mains by means of fire water pump stations, delivering approximately 7500 l/min @ 850 kPa, designed and installed in accordance with ASIB 12th edition rules. The pumps will draw water from dedicated fire water reservoirs and will have a total combined volume of around 675 m^3 of effective capacity, equally divided into 2 sections.

The installation will comprise of a main electric pump, a standby diesel driven pump of the same duty and a jockey or make-up pump used in order to maintain the system's pressure in the event of small usage and/or a leak. The pump system will distribute the fire water to the required areas via a number of ring main systems. Typical distribution points include:

- Strategically placed fire hydrants,
- Deluge water spray systems on conveyors,
- Deluge water spray systems on transformers, hydraulic power packs,
- Sprinkler systems within buildings, and
- Hose reels within infrastructure buildings.

The following is further allowed for in the capital cost in terms of fire systems:

- All fire panels will be fitted with its own power supply, with an incoming power requirement of between 100 v and 240 vAC as well as a battery back-up.
- All surface sub-stations and MCCs will be fitted with fire detection and a clean agent gaseous suppressions system. The gaseous suppression system designed around is typically Pyroshield which is readily available and spare cylinders can be kept on site and swopped out as required. Each fire detection panel will have potential free contacts available to report to the PLC, the current status of the panels. AC / Incoming power shutdown contacts will also be made available at the panel and in the event of a fire condition will initiate shutdown of the incoming power and AC equipment in order to ensure that the gas is contained within the protected area.
- Where required, sprinkler installations will be installed in areas where capital equipment is either stored or maintained. With regards to general infrastructure buildings, will be protected by extinguishers, hose reels as well as fire hydrants located in close proximity.
- All production critical conveyors will be protected by means of deluge systems. The elevated portions of production critical conveyors will be protected as well as all drive stations, heads, tails and load points.

18.1.15 Construction Facilities

Where possible permanent infrastructure will be utilised to minimise temporary construction facilities. Temporary construction facilities and services has been allowed for during the construction phase of the project, where permanent infrastructure has not been constructed or allowed for.

These facilities are a crucial part of construction management, as sites can be very demanding involving the co-ordination and movement of large quantities of materials as well as high-value products, plant and people.

Effectively and accurately laying out a site ensures that the construction works are undertaken safely and efficiently. Correct sizing and positioning of temporary facilities can help to reduce travel times, congestion and waiting times, making the site a more effective workplace.

The following temporary utilities has been allowed for:

18.1.15.1 Accommodation

Allowance for construction accommodation has been made at the following three locations:

- Existing Kamoa Camp: The existing Kamoa camp caters for 350 people. 100 of these accommodation facilities are permanent and 250 are temporary. After construction is completed, only the permanent accommodation will be utilised.
- Kakula Main Camp: This camp will cater for 1600 personnel in total of which only 1000 will be constructed for the first 3 Mtpa plant in phase 1. This accommodation will cater for the client team, mining contractors and EPCM contractor.
- Contractors Camp: An area is allocated and fenced off to cater for the main contractors (Earthworks, Civils, EC&I, SMPP and Building works contractors). This camp will cater for 1000–2000 people at a time. The accommodation cost has to be covered in the contractors cost.

18.1.15.2 Construction Power

All construction power is to be supplied by local diesel generators provided by the contractors as required.

Provision has also been made to reticulate grid power to the construction offices, the construction laydowns and the concentrator plant construction areas from a series of mini-substations. Remote areas, for example the tailings facility, will continue to be supplied by local diesel generators as required.

18.1.15.3 Construction Water

Initially the bulk water supply, for construction of the mine, will be provided by local groundwater from boreholes. The water will be transported to site with water bowsers and discharged where required.

18.1.15.4 Other Construction Services

In addition to construction accommodation, power and water supply, the following is further allowed for:

- Construction Laydown: Area has been identified to be used as the main construction laydown areas, where all surface contractors will establish their main construction camp.
- Construction Office: The Main office block will be utilised in the short term as construction offices and will be extended when needed.
- Construction Stores: A portion of the permanent Capital and Main store will be constructed and utilised during the construction as storage space for equipment.
- Construction Communication: An independent IT network in the construction offices with Internet Data communications (by Satellite), is catered for the EPCM project team. Handheld mobile radios and two base stations are provided for site voice communications.
- Construction vehicles has only been allowed for the EPCM Project team. Existing project vehicles will be utilised.
- Construction SHEQ: Allowance were made for lighting detection equipment, breathalyser's, first aid equipment, permit and inspection books, construction office firefighting equipment, working at height rescue kit, PPE and environmental spill kits.
- Construction Signs: Allowance was made for standards signs required for a construction site.
- Construction Ablution Facilities: Ablution facilities during construction is each contractor's responsibility.
- Construction IT and Computer Equipment: Allowance was made for an independent IT network including file server, multifunction printer/copier, A0 plotter and WIFI networking within the construction offices. Office personal computers, audio visual projectors and a video conferencing facility is also catered for.
- Construction Access and Security Facilities: Temporary container type units have been allowed to assist with access control of employees to the site. Vehicle access to site will be controlled by boom gates.
- Construction Waste Facilities: Allowance was made for a construction salvage yard and waste skip laydown areas.

18.2 Kamoa Site Infrastructure

18.2.1 Introduction

This section has not been changed from the Kamoa 2017 Development Plan 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 Kamoa 2017 Development Plan.

This section describes the project infrastructure work that was developed for the Kamoa 2017 PFS. The project infrastructure includes power supply, tailings dams, communications, logistics, transport options, materials, water and waste water, buildings, accommodations, security, and medical services.

18.2.2 Site Plan and Layout

A plan showing the locations of the mines and key infrastructure for Kakula and Kansoko mines is shown in Figure 18.1. Figure 18.16 displays the locations of the proposed Kamoa plant site, and closely associated facilities.

Figure 18.16 Site Conceptual Infrastructure Layout Plan

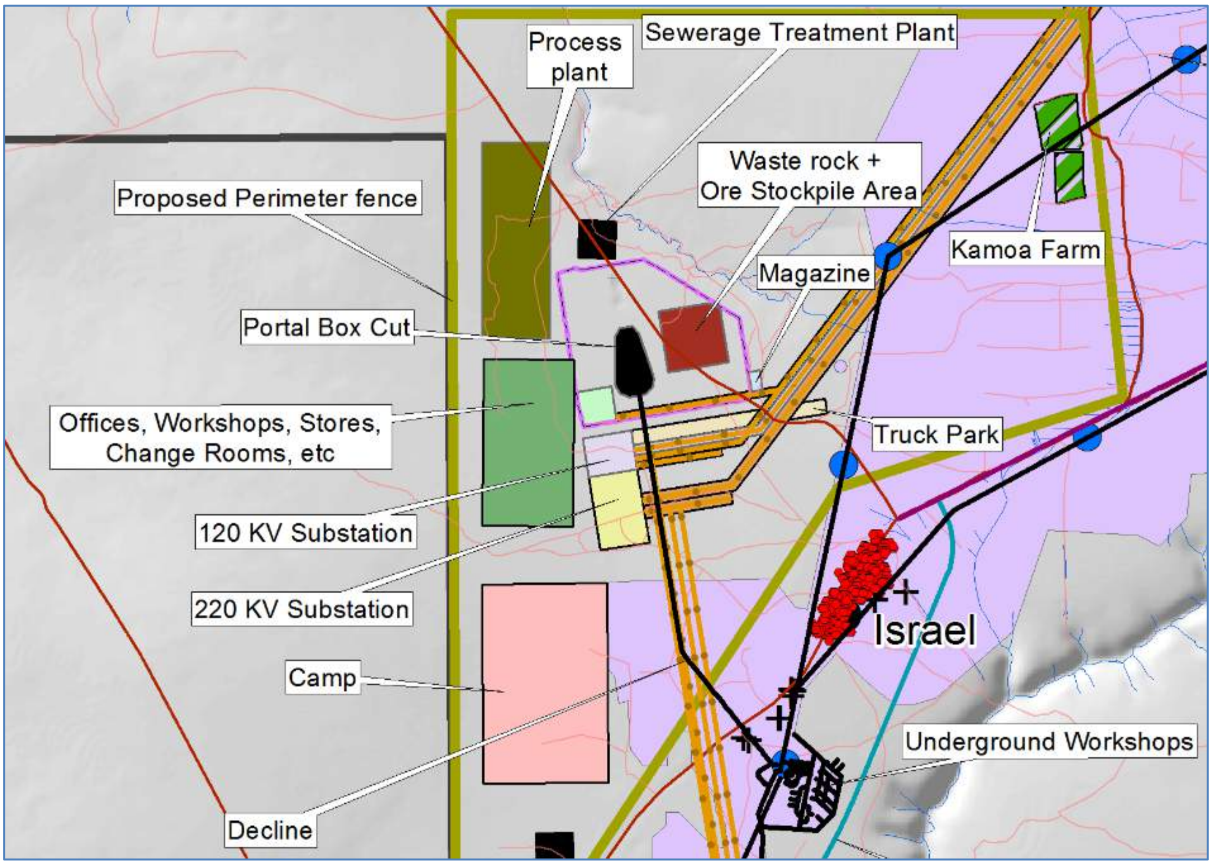


Figure by Ivanhoe, 2017.

The site is compact and incorporates the process plant, utilities, reagent preparation, laboratory, offices, construction camp, electrical infrastructure, water infrastructure, surface mining offices and workshops, vehicle parking, warehouse storage, concentrate storage and lay-down facilities. All infrastructure has been incorporated in the capital cost estimate.

18.2.3 Power

18.2.3.1 Bulk Power Supply

Refer to Section 18.1.4.2 for power supply to Kamoa. For construction power (10 MW), a 120 kV high-voltage spur line (20 km long) has been built to tap power from the RO-Kisenge line to the Kansoko Mine. RO is the acronym for "Répartiteur Ouest" i.e. Western Dispatch substation in Kolwezi. A 120/11 kV, 15 MVA mobile substation has also been installed and commissioned to feed construction power. The line and substation will be retained as emergency back-up power supply after the commissioning of the main 220 kV supply line and substation. Diesel generators for back-up power have been installed and are operational. The diesel generator capacity will be increased in size to ultimately provide the mine and plant with the required standby power.

18.2.3.2 Transmission and Substations

The power plants substations and lines will be refurbished. A new Gas Insulated Substation (GIS) 120/6.6 kV substation will be built at Mwadingusha hydro power plant. Koni and Nzilo hydro power plants substations will be refurbished completely. OPGW (optical ground wire) will be installed to the two Nzilo-RO 120 kV lines (20 km).

The two 120 kV bays at RC substation in Likasi where Koni and Mwadingusha power connect to the SNEL grid will be also refurbished.

In the interim or first phase, 10 MW can be supplied to the Project over a new 20 km transmission line from the RO-Kisenge line to Kansoko Mine for construction power, through a 120/11 kV, 15 MVA mobile substation that is installed at Kansoko Mine.

At the second phase, in order to achieve high power availability over the longer-term, a new double 220 kV circuit transmission line (20 km) will be constructed to feed power to the 220/11 kV Kansoko Mine substation. A new SNEL 220 kV sub-station (named NRO, Nouveau Répartiteur Ouest) will be constructed adjacent to the existing 220 kV substation owned by Sicomines mine west of Kolwezi. NRO will be fed from the SCK substation in Kolwezi, which is a major transmission hub in SNEL's southern network, connecting to the Northern network and the Inga power plant via a 1,200 km DC line. The Figure 18.17 shows the design of new transmission lines and substations.

Figure 18.17 Planned Transmission Lines and Substations

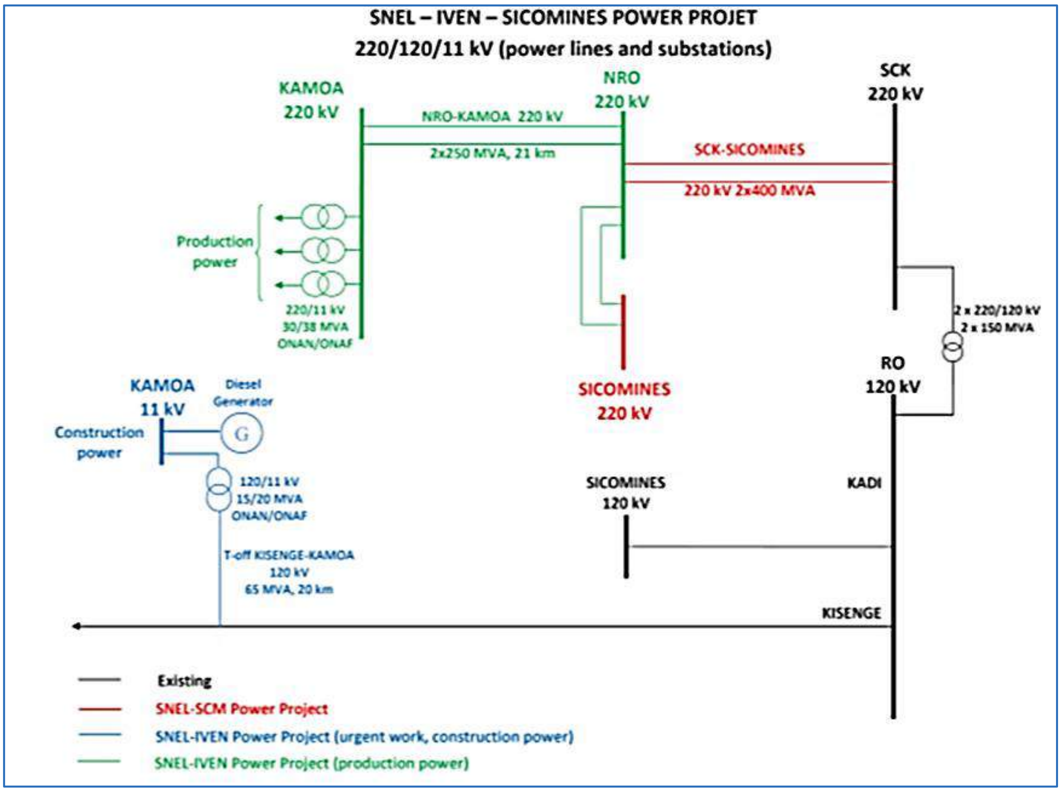


Figure by Ivanhoe, 2016.

18.2.4 Tailings Storage Facility

Epoch Resources (Pty) Ltd (Epoch) completed a basic design of the Tailings Storage Facility (TSF) and associated infrastructure as part of the Kamoa 2017 PFS.

The terms of reference that Epoch was responsible for include:

- A Tailings Storage Facility (TSF) that accommodates 109,480,000 dry tonnes of tailings over a 26-year LOM.
- A Return Water Dam (RWD) and Return Water Sump (RWS) associated with the TSF.
- The associated infrastructure for the TSF (i.e. perimeter slurry deposition pipeline, stormwater diversion trenches, perimeter access road etc.).
- Revalidation of the Mupenda site for the higher production rate (i.e. 6 Mtpa mill feed).
- Estimation of the capital costs to an accuracy of ±25 percent, operating costs associated with these facilities to an accuracy of ±25 percent and closure costs to an accuracy of ±35 percent.
- Estimation of the costs over the life of the facility.

The site selection study undertaken, by Epoch, found the most favourable site as being the Mupenda site.

The key design features of the TSF are as follows:

- The TSF will be constructed as a double valley impoundment dam with a compacted earth impoundment wall. This will have the following features:
  - The TSF impoundment walls will be constructed as a downstream facility.
  - The wall is to be raised in 7 phases, where Phase 1 is at elevation 1465 mamsl and the last phase is at elevation 1495 mamsl.
  - A final phase (Phase 8), comprising a smaller upstream impoundment wall will be constructed on top of Phase 7 when the rate of raise reduces to <1.0 m/year.
  - The TSF has a total footprint area of 540 Ha, a maximum height of 52 m and a final rate of rise of <1.0 m/year.
- A Return Water Dam with a storage capacity of approximately 45,000 m^3.
- A concrete lined Return Water Sump with a water storage capacity of 2,000 m^3.
- A slurry spigot pipeline along the crest of the TSF.

18.2.4.1 Project Location

The terrain is mostly grasslands with some dense pockets of trees. The general topography of the Mupenda site area can be seen in Figure 18.18.

Figure 18.18 General Topography of the Preferred Kamoa TSF Area

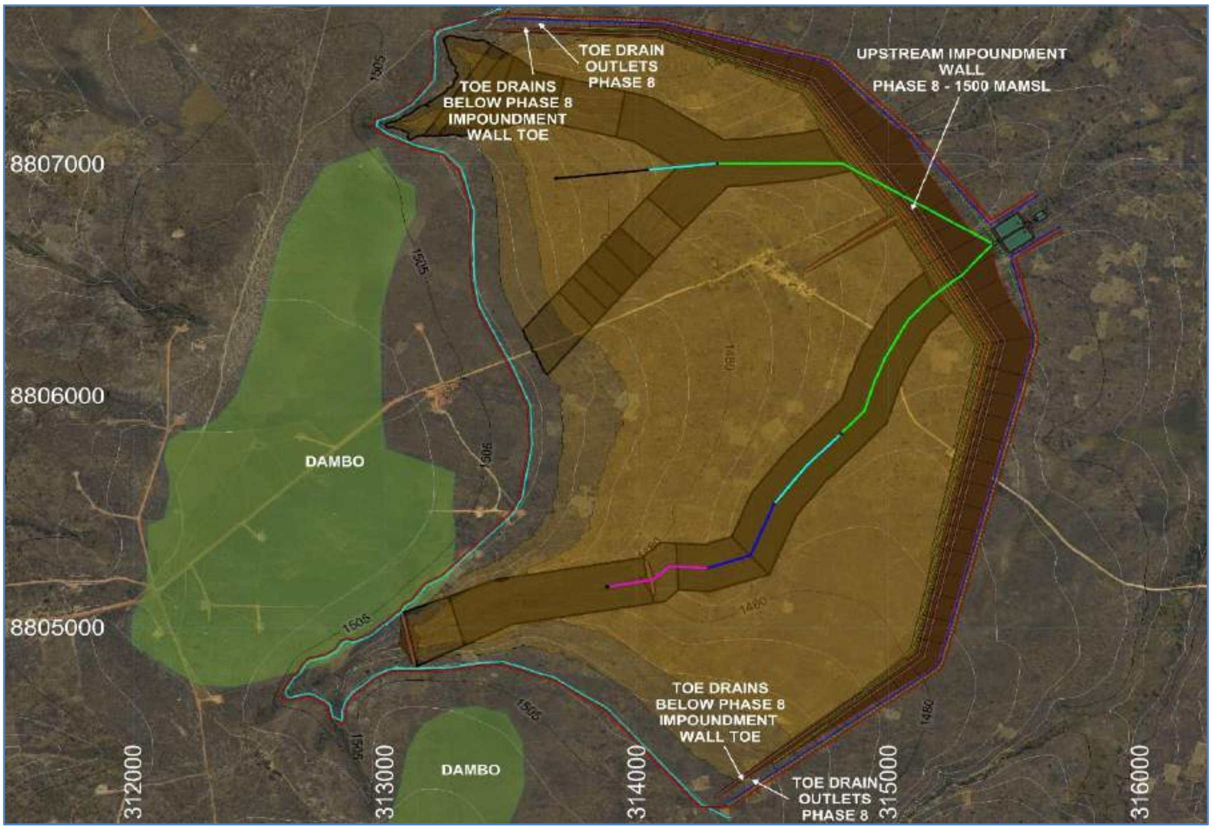


Figure by Epoch, 2017.

18.2.4.2 Design Criteria and Assumptions/Constraints

The design of the TSF was based on the design criteria shown in Table 18.5.

Table 18.5 Design Criteria Associated with Kansoko TSF

Description	Value	Unit
Design Life of Facility	22	years
Average Tailings Deposition Rate	5,520,000	Dry tpa
Particle SG of Tailings product	2.85	
In-situ Void Ratio	1.0	
Particle size distribution of Tailings product	80% passing 53 micron	
Average dry density of tailings	1.4	t/m^3
Site's Seismicity	0.08 g	PGA
Design Storm (24hr, 1 in 100 year)	139	mm

18.2.4.3 Liner Requirements

Golder's geotechnical division was tasked with assisting with the preliminary specification of the liner solution at the Mupenda TSF site. This was documented in their report titled: "Geotechnical and Geochemical Aspects of the Liner Recommendation for the Mupenda TSF". They identified a layer of Kalahari sand along the valley which does not provide the necessary permeability requirement to adhere to the DRC Regulations. This sandy layer was found to have a permeability higher than the limit of 1x10-6 cm/s. The remainder of the footprint was found to adhere to this permeability limit. Therefore, only the area underlain by Kalahari Sands will require a liner.

18.2.4.4 TSF Site Selection

The preferred TSF site is the Mupenda site. The site selection study was performed in 2014 by Epoch and documented in their report titled: "Site Selection Report – Kamoa Copper Project, TSF for a 3 Mtpa Plant – Addendum". This site was chosen for the TSF for the following reasons:

- The topography and soil properties are such that it will not require expensive measures to both contain the tailings and prevent ground water and surface water contamination.
- The risks associated with this site were deemed the lowest out of the other options due to the contaminated catchment downstream of the TSF due to the historical failure of the Potopoto TSF, as well as little to no people residing downstream of the site.
- Lowest construction cost of all the options considered.

18.2.4.5 Design Considerations

The design of a TSF usually begins with determining what type of facility will be selected to contain the tailings. Two common types of facilities are self-raised (upstream) and full containment (downstream). Due to the fineness of the overflow tailings to report to the TSF, it will not be possible to self-raise the tailings. Therefore, a full containment facility, with a downstream construction methodology was selected. The fineness of the Kakula tailings also has the following implications:

- Subsoil drains cannot be constructed in the tailings as they will blind and become inoperable. A full containment facility can utilise a curtain drain to reduce the phreatic surface through the wall. A curtain drain will not be in contact with the tailings and thus cannot blind.
- Return water will contain suspended solids if it is not allowed sufficient time to settle. This can be mitigated by constructing a silt trap/settling facility or maintaining a pool for longer durations on the TSF.
- The beach slope of the TSF is expected to be very flat, making pool control very difficult for an operator.

18.2.4.6 Stage Capacity and Site Development Strategy

The impoundment wall has been phased in order to delay capital expenditure as far into the life of the facility as possible. In order to effectively phase other construction items (such as the penstock pipeline and liner), one intermediate wall which will hold the tailings back while allowing the contractor to construct and install the penstock and liner, as well as a final back wall, will be provided at the upstream side of the southern valley. The penstock, liner and intermediate and back walls will have several stages which correspond to certain impoundment wall phases (see Figure 18.19).

Figure 18.19 Impoundment Wall and Self-Raise Lift Phasing

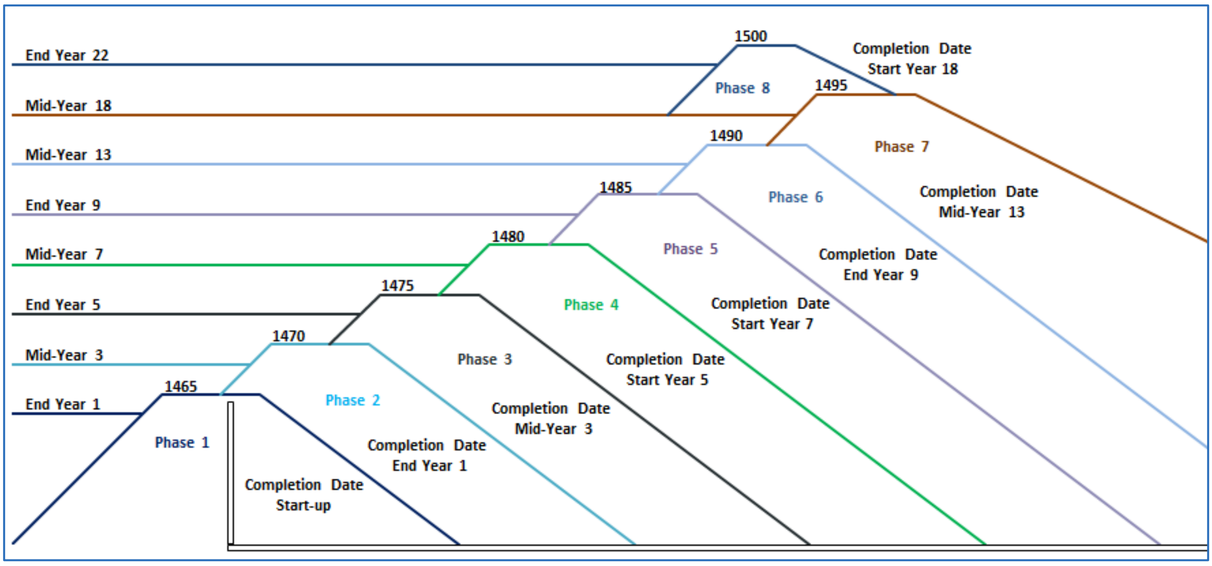


Figure by Epoch, 2017.

The final lift of the wall would be constructed as an upstream wall as shown. In order to confirm whether this is feasible, stability modelling of this option must be undertaken, as well as field investigations during operation of the facility.

18.2.4.7 TSF Construction Works

The construction of the TSF wall would include the following:

- Topsoil stripping to a depth of 300 mm beneath the TSF footprint.
- A box-cut to a depth of 500 mm beneath each impoundment wall.
- A compacted key below the Phase 1 impoundment wall;
- A compacted earth starter wall with the following dimensions.
- 17 m high (i.e. crest elevation of 1465 mamsl).
- 15.0 m crest width.
- 1V:1.5H upstream side slope.
- 1V:2H downstream side slope.
- A Curtain Drain inside the impoundment wall, to reduce the phreatic surface through the wall;
- A stormwater run-off trench and berm around the TSF from which water is directed away from the TSF;
- A stormwater diversion channel with its associated cut-to-fill berm;
- A buried 900 ND Class 150D spigot-socket precast concrete penstock pipeline in each valley, composed of single intermediate intakes and a double final vertical 510 ND precast concrete penstock ring inlet.
- A 1500 micron liner along the bottom of each valley and approximately 200 m wide, in order to prevent tailings water seeping through the highly permeable Kalahari sands.
- A 280 ND slurry spigot pipeline along the length of the TSF perimeter; and
- A two-compartment reinforced concrete RWS.

The specified size of the penstock pipeline and the slurry delivery pipeline has been based on preliminary design calculations and should be re-evaluated during the next phase of the project.

18.2.4.8 TSF Depositional and Operational Methodology

The depositional technique selected for this project will be a valley impoundment, hydraulically deposited spigot facility. The impoundment wall will be constructed using waste rock or borrow material and tailings will be deposited behind the wall and into the valley. This design is a common construction technique used in tailings storage facilities. The three principal designs are downstream, upstream and centreline structures, which designate the direction in which the embankment crest moves in relation to the starter wall at the base of the embankment wall. The Kamoa TSF is a downstream structure. The tailings are usually discharged from the top of the dam crest creating a beach and a resulting supernatant pool develops as far away from the wall as possible. Where the tailings properties are suitable, natural segregation of coarse material settles closest to the spigot and the fines furthest away.

18.2.4.9 Water Balance

An overall water balance model for the TSF has not been undertaken during the PFS phase but should be considered during the next phase of the project. Based on Epoch's experience it is expected that the equivalent of 50 to 60% of the water in the tailings slurry will be recovered and returned to the plant as an annual average. A large amount of the tailings water will evaporate and be held in the dam with the settled solids, but rainfall will replace some of this lost water.

18.2.4.10 Closure Activities at Cessation of Operations

At the cessation of operation of the TSF, the focus will be on the cover and vegetation of the top surface of the facility, the decommissioning of facilities associated with the TSF and the construction of storm water and erosion control measures as required. The duration of the final closure process may be affected by the length of time required for the basin of the facility to dry sufficiently to enable the placement of cover material in preparation for the vegetation establishment.

18.2.4.11 Risks

The possible project risks associated with the current TSF design are as follows:

- Kamoa is situated in a seismically active area. No stability analyses have been undertaken for the TSF to confirm that the current TSF geometry will withstand a seismic event.
- A suitable borrow pit has not yet been identified for use in the impoundment wall.

18.2.4.12 TSF Recommendations

For the Definitive Feasibility Study stage of the project, it is recommended that the following be included:

- A more thorough geotechnical investigation of the TSF site in order to confirm the type, extent and characteristics of the in-situ materials as well as available construction materials.
- A more thorough water balance study for the TSF be undertaken.
- A seepage analysis and slope stability study be undertaken to confirm the seepage regime through the TSF as well as to confirm the TSF stability during a seismic event. The results of these analyses could impact greatly on the geometry of the TSF walls and ultimate height of the facility.
- Confirmation of the physical characteristics of the tailings product based on laboratory testing of a representative sample generated by the IFS4a flotation testing flow sheet. This must include flume and rheology tests to determine the tailings beach slope.
- An assessment of the need for additional contamination control measures such as HDPE liners or clay liners, dewatering and/or contaminated water treatment.

- Possible further optimisation of the TSF preparatory works in terms of layout, footprint extent, etc.
- Compilation of a more detailed schedule of quantities describing the proposed preparatory works and the pricing of the schedules to a greater level of accuracy.

18.2.5 Site Communications

Communication to the site is currently provided by high-bandwidth satellite internet connection provided by O3B with a Vodacom cellular data internet connection for back-up. Fibre optic internet service providers are operating in Lubumbashi 300 km from Kolwezi and there are reasonable prospects for this to be extended to Kolwezi and Kamoa in the near future.

A fibre optic network has been installed across the site for the existing temporary facilities and this will be expanded as the permanent facilities are constructed. Cell phone coverage is available on site from Vodacom and Orange cellular providers. Radio systems are already operational at Kamoa and these will be expanded on surface and underground as the project is developed.

18.2.6 Site Waste Management

Currently land fill sites or waste collection facilities in the Kolwezi area are limited. There are hazardous waste management contractors or services based in Kolwezi that can deal with oils, batteries, bio hazardous waste etc. There are a number of companies collecting used oil for recycling and for use as burner fuel. Kamoa plans to construct a landfill site near the mine for non-hazardous waste disposal. A suitable site has been chosen for this and a concept design and costing for this has been prepared by Golder Associates.

An integrated approach to waste management for the project will be needed. This would involve reduction, reuse, recycling and would be done onsite through waste separation. Some of the methods incorporated would be through composting, alternative uses based on stockpiling areas and storage for other disposal (for hazardous chemicals like oils, batteries, vehicle filters and old parts etc.). This approach will be developed further during the feasibility phase.

18.2.7 Roads and Earthworks

18.2.7.1 Roads

The following facilities have been allowed for inside the plant and mine area:

- Plant roads. All plant roads will be gravel roads.
- Plant to portals roads. A 6 m wide gravel road will be provided.
- Plant to tailings storage facilities. A 6 m wide gravel road will be provided.
- Service roads (conveyor, ventilation fans, slurry pipelines). 4 m gravel roads will be provided as serviced roads.
- Village access road. A 6 m gravel road will be provided.
- Village roads. Varying road widths will be provided, depending on the hierarchy of the road in the village. All roads will be surfaced roads.

18.2.7.2 Terracing and Earthworks

Terracing shall be designed with suitable grading for efficient draining of stormwater run-off and keeping in mind optimisation of cut-and-fill earthworks quantities. Stepped terraces shall be proposed to accommodate mechanical and process requirements on the plant. The Kamoa site has been identified to consist of collapsible soils of low bearing capacity that will not provide adequate support for heavy structural foundation loads. Therefore, terrace layer works shall be designed for removal of unsuitable in-situ soil and backfilling with structural fill layers to provide a stable founding medium for structural foundations to carry heavy mechanical and process equipment. For major foundation loads such as the ball mills, piling will be required. All topsoil will be stripped from terrace areas and stockpiled for use during site rehabilitation.

18.2.8 Logistics

Refer to Section 18.1.9.

18.2.9 Airports

Refer to Section 18.1.10.

18.2.10 Water and Wastewater Systems

18.2.10.1 Water Demand

The estimated water demand for the project scenario is given in Table 18.6. These figures are an average through the year. There will be a large variation between dry and wet seasons. A contingency has been added to account for unanticipated consumption, such as increased tailings dam water retention due to finer tailings P_{80}.

Table 18.6 Estimated Water Demand

Description	Units	Quantity
Mining Water Requirement	m^3/day	320
Concentrator Water Requirement	m^3/day	7,600
Potable Water Requirement	m^3/day	280
Contingency	%	10
Total Daily Requirement	m^3/day	9,100

Raw water will be provided to the site via the four production boreholes forming the Southern Wellfield, as identified by Kamoa. The boreholes will be connected to a common overland pipeline (7 km) which will feed into a water storage dam located at the plant. This will provide all necessary raw water which will then be used to provide the required process water makeup, gland water, fire and reagent make-up water. Most water loss is due to evaporation and seepage from the TSF. It is estimated that the equivalent of 50%–60% of water going to the TSF will be returned. A return water pipeline (10 km) will bring water from the TSF to the process water tank. Water from mine de-watering will also be utilised for process water make-up.

18.2.10.2 Bulk Water

The assessment of the bulk water supplies has been undertaken with the view of supplying the estimated water demand of 9.1 ML/d.

Two potential sources have been identified for the bulk water supply. The first is the aquifer within the sandstone forming the Kamoa and Makalu Domes, and also constitutes the footwall to the mining operations. The second potential source is the major rivers within the Kamoa exploitation licence, including the Lulua, Tjimbudgi and Lufupa rivers. The rivers have strong flow year-round and sufficient water could be extracted with a simple weir arrangement.

River water is considered a contingency at this stage, since it is estimated that sufficient water will be available from boreholes and mine de-watering. The bulk water supply will be obtained from the 4 boreholes (3 production and 1 standby holes) forming the Southern Wellfield. This supply will be augmented by water obtained from the decline dewatering boreholes.

The bulk water supply could be augmented by groundwater inflow into the underground workings. The volume of mine water inflow will be determined in the future.

According to the DRC Mining Code, an exploitation licence gives the holder automatic rights to use the surface and ground water on the licence area, so there is minimal permitting risk for use of this water.

18.2.10.3 Potable Water

Potable water for mining, ablution facilities, kitchens and emergency stations (eyewash and showers) will be obtained from boreholes and treated by means of disinfection only (chlorination). An appropriate drinking water standard will be applied, referencing indicators such as bacterial content, residual chlorine, turbidity, and dissolved solids. The borehole water at Kamoa is very good quality, with exceptionally low dissolved solids levels.

Potable water will be distributed via pipe racks and sleeper ways along with other services where possible and underground as necessary.

18.2.10.4 Stormwater Infrastructure

The Department in Charge of the Protection of the Mining Environment in the DRC requires that an Environmental Impact Study (EIS) is performed for any proposed mining activity within the DRC. The EIS is prepared using the Mining Regulations, Annex IX (Walmsley, B. & Tshipala K.E., 2012). Article 19 of Annexure IX requires that all mines develop measures to reduce the inflow of uncontaminated run-off water into the mining site water management system. Article 82 of Annexure IX requires that the sizing of any water retention structures accommodates for the water contribution resulting from a projected 24-hour flood with a return period of 100 years. The sizing of the stormwater management plan, the pollution control dams and the pipelines with their required pumps are all based on these regulations.

18.2.10.5 Stormwater Management Plan

The assumptions made for this investigation include:

- Due to the lack of sufficient data closer to the Kamoa site, the Solwezi rainfall data was used to analyse the one in 100-year return period 24-hour rainfall event.

The stormwater management plan and pipeline system were developed based on the most current site arrangement information available to Golder Associates Africa.

The location of the plant, stockpile, and decline area is shown in Figure 18.20.

The run-off from this area will be contained with earth dams and will need to be managed within the mine’s dirty water system. Berms are required around the perimeter of the area to prevent run-off from the upslope areas entering the site. The run-off from the site is collected in berms/channels located on the northern perimeter of the area. The run-off collected by these berms is directed to a stormwater control dam located to the north of the site. The capacity of the stormwater control dam is sized to store the run-off volume from the 100-year 24-hour storm event.

The 1:100-year 24-hour storm depth of 139 mm, calculated using the daily rainfall data measured at the Solwezi rain gauge, was used to calculate the run-off volume that would report to the stormwater control dam. The run-off from the catchment for the 100-year event will not be 100%. There will be losses both from depression storage and infiltration. The SCS technique was therefore applied to calculate run-off from this event. Based on a catchment area of 66 ha, a flood volume of approximately 58,000 m^3 was estimated for the 100-year 24-hour event. This capacity is therefore recommended for the stormwater control dam.

The area of the stormwater dam is 1.5 ha, with a 4 m depth. The dam is assumed to be a cut-and-fill dam with the wall material sourced from the dam basin. Geotechnical studies will be required to confirm the suitability of the materials for dam construction. The required lining for the dam will be determined during the EIA, but allowance for a liner in the costing is included at prefeasibility stage.

The stormwater management plan included in this document is done at a high level and should be considered a conceptual plan. A more detailed stormwater management plan and pipeline system will be developed as the mining project progresses.

Figure 18.20 Stormwater Dam

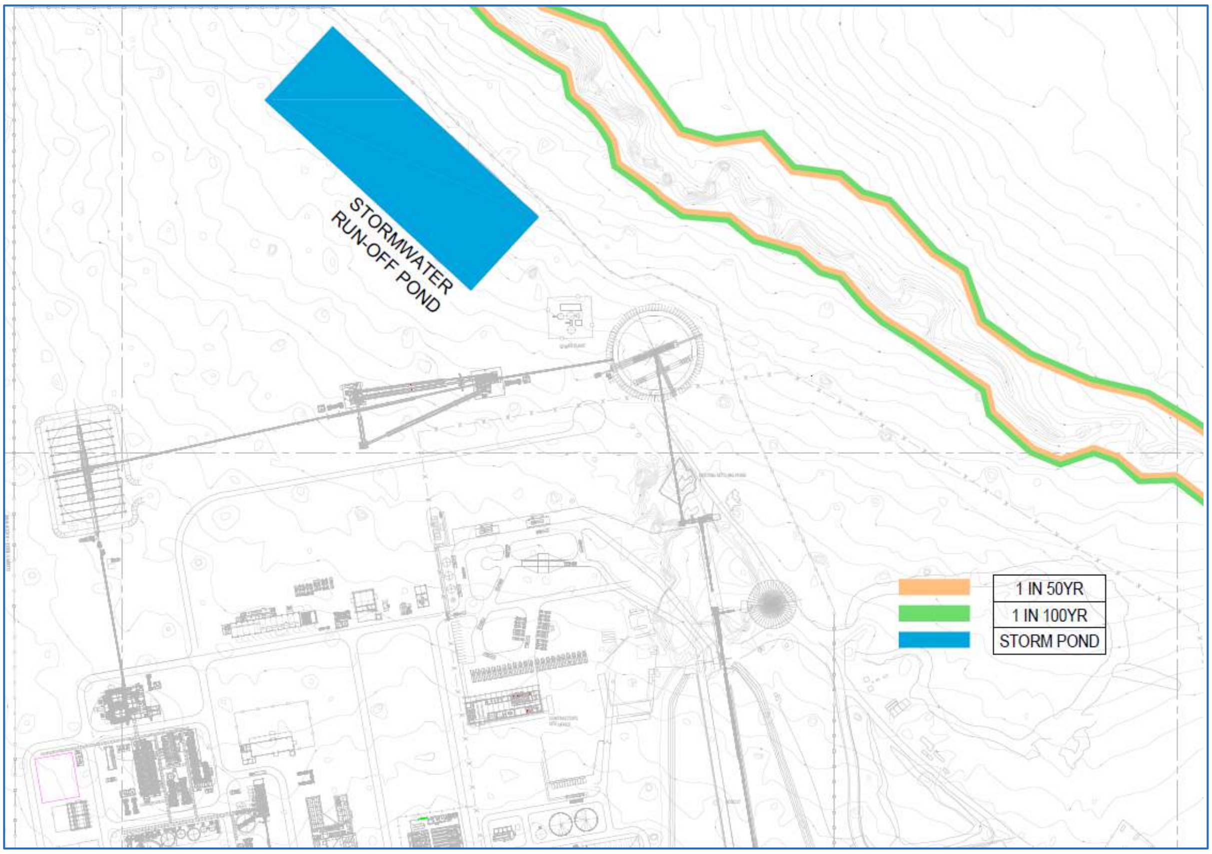


Figure by MDM, 2017.

18.2.10.6 Wastewater

Sewage from kitchens, laundries, and ablutions will drain via underground sewers to a sewage treatment plant and will be treated to produce an effluent of a suitably safe standard for process use.

Floor washings that contain organic contaminants, from kitchens and ablution blocks, will also drain via the sewers to the treatment plant. Floor washings that are potentially contaminated with mineral oils (workshops, refuelling and lube and diesel storage areas) will drain to the run-off dam.

Kamoa currently has a sewerage plant for the existing accommodation camp and similar plants will be utilised at the mine site and future accommodation camp. These plants are zero-sludge plants, fully digesting solids into solution. The treated water would be used for irrigating gardens or be recycled to the concentrator process plant. Other wastewater streams and by-products such as acid are covered under plant process design.

18.2.10.7 Potential Water Treatment

It is predicted that during the initial stages of mining, all excess water will be re-used at the plant as make-up.

However, as mining progresses with bigger voids forming, larger volumes of ground water could be expected within the underground workings, which will require dewatering.

The mine water is not expected to be acidic. Initial treatment will largely involve settlement, removal of oil and grease, etc. High concentrations of nitrate may also have to be removed as well as any heavy metals.

However, as the water balance shifts to positive over the LOM, including seasonal fluctuations, the acidity of the water could increase, necessitating treatment by installation of a water treatment plant.

A high-level capital cost estimate for a 1 ML/day plant to address acidity, presence of metals and salts in the mine water will amount to approximately US$1M.

The cost for a water treatment plant could be either provided for through the contingency provision for the project or from the closure cost provision for the Mine, especially in the event that water treatment is required beyond closure.

An option for treatment of excess waste water is to use evaporative mist sprays over the TSF. Much of the water evaporates from the mist and the remaining water containing dissolved salts and solids falls into the TSF. This method is successfully being used at a number of other mines in the region. However, it can only be used during windless and sunny periods.

18.2.11 Fire Protection and Detection

The fire protection and detection system for the surface plant and infrastructure (excluding all underground mining which is covered separately) will be developed in consultation with and subject to final approval from the Owner's risk assessors. The system will be designed to comply with DRC legislation (where applicable), the project Health and Safety standard/s, project specifications and fire protection standards as adopted by the Project.

The development of the fire protection and detection system will take into account all high-risk areas of the plant, as these may require specialised fire systems. The system will include a combination of passive measures (e.g. fire walls, physical isolation etc.) and active systems (e.g. fire detection, fire water systems, gas suppression systems, etc.).

Fire detection equipment will include a Fire Indicator Panel (FIP) located in the main control room area, and local intelligent Sub Fire Indicator Panels (SFIP) as required located around the site.

The fire detection system will be independent of the Process Control System (PCS) and will be specified as part of the overall Fire Protection System for the plant, which will also include the Fire Water System, Gas Suppression Systems and any other specialised systems (if required for high risk areas).

Fire water storage will be a dedicated water supply volume, sized in accordance with the requirements of the applicable fire standard. The fire water pump house will be designed with a high degree of reliability, and would typically include a jockey pump (to maintain system pressure under normal non-fire conditions), as well as electric and back-up diesel fire water pumps.

The water supply will be sized to provide the required maximum firewater flows for any single fire event. Fire water will be distributed around the plant via a fire water reticulation network, which will connect to strategically placed hydrants, hose reels, sprinkler systems, deluge systems, and / or foam systems as required.

Buildings and offices will be equipped with hose reels and portable extinguishers, in accordance with the governing building standards and project specifications.

Gas suppression systems will typically be used for critical areas such as electrical rooms, control rooms, server rooms etc. Hand-held extinguishers will be distributed around the plant and in all buildings.

The size of the site will require the availability of at least one fire fighting vehicle (with 4 x 4 capabilities) to ensure it is available to deal with fire events in remote areas of the site.

18.2.12 Hospital and Medical Facilities

The clinic and first-aid facility will be housed together at a suitable position near the main gate. The clinic will be suitable for all occupational health checks, regular consulting rooms, emergency trauma rooms, and 8 hospital ward beds for overnight patients. A separate first aid facility at the mining control room has been included for dealing with mine accidents. Medical equipment, including an ambulance, will be provided. Medical evacuation for ex-patriot employees will be provided by an outside contracting service. Hospitals are available in Kolwezi for employees and their families resident in Kolwezi.

18.2.13 General Building Requirements

The surface building requirements were obtained from other projects with the similar number of personnel, fleet size and production rates. An all-inclusive rate per square metre of floor area was applied for steel and brick structures. The estimate includes furniture, fitting, electrical appliances, power supply infrastructure and communication. The estimate is based on contractor construction. It is planned to erect some of these buildings early in the construction period so they can be used during construction, thereby minimising the requirement for temporary construction buildings. The buildings are described in the following sections.

18.2.13.1 Concentrator Buildings

- Administration building/offices (522 m^2).
- Clinic and first aid station (600 m^2).
- Kitchen /canteen (442 m^2).
- Change house (344 m^2).
- Weighbridge control room (22 m^2).
- Laboratory (960 m^2).
- Gatehouse and security (277 m^2).
- Training Centre (1,200m^2).
- Mess Complex (450 m^2).
- Satellite ablutions (25 m^2).
- Substations.
- Explosives storage (bunding allowance for mining explosives handling).
- Plant and Vehicle workshop (light crane loads) (433 m^2 and 300 m^2).
- Concentrate Handling (5,720 m^2).
- Plant stores (933 m^2).
- Plant control room (126 m^2).
- Reagents store (672 m^2).

18.2.13.2 Mine Surface Buildings

- Aggregate and multipurpose store (281 m^2).
- Briefing Area (400 m^2).
- Capital store (536 m^2).
- Change House Complex (3,801 m^2).
- Engineering Workshops (719 m^2).
- Firewater pump station (43 m^2).
- Medical room (77 m^2).
- Mine Rescue Room (77 m^2).
- Shaft Control Room (389 m^2).
- Shaft Gate House (275 m^2).
- Shaft Offices (1,419 m^2).
- Surface gas store (70 m^2).
- Surface lubricant store (39 m^2).
- Surface paint store (39 m^2).
- Tyre store (434 m^2).
- Warehouses (Stores) (1,417 m^2).

18.2.14 Owner's Camp

18.2.14.1 Accommodation

A permanent village will be constructed at the existing exploration camp location to provide accommodation for owner's team management, expatriates and EPCM consultants. Single units will be two bed with en-suite bathroom and family/executive units with 3 bed, two bathrooms with open plan living room and kitchen.

18.2.14.2 Facilities

The following facilities will be included:

- New kitchen and mess complex.
- Recreation centre.
- Sports facilities.
- Administration offices.
- First aid room.
- Laundry.

Wireless internet and cable TV in all rooms will be allowed for.

18.2.14.3 Roads and Services

The following roads and services will be provided in the accommodation area:

- Perimeter security fence.
- Gravel access roads to housing units.
- Parking (remote from rooms).
- Water reticulation, sized for fire flows and provided with hydrants.
- Sewer reticulation and treatment.
- Internal communications.

18.2.15 Construction Facilities

To facilitate the execution of the project, various temporary facilities are required. These facilities include:

- Construction Camp: A 1500 bed construction camp to accommodate the construction workers during execution will be erected within walking distance of the mine. The camp plan assumes single-quarters accommodation and will include bedrooms, ablution facilities, dining area and kitchen, recreation area as well as admin offices and guard house. Services such as water, sewer, electricity, in-room wireless internet and TV will be provided. As the camp will be used during the operational phase for on-shift accommodation it will be built for a 25-year service life. All ablutions will be shared. Seniors will each have their own rooms and Juniors will be two per room.
- Construction Site Offices: The Concentrator and Mining infrastructure buildings will be erected to be utilised as construction site offices. Once construction has been completed, the buildings will be refurbished and handed over to operations.
- Laydown areas: Contractors will need prepared areas to establish their site offices and areas to store construction material, equipment and vehicles. Fenced terrace areas with water, sewer and electrical connections will be provided.
- Customs Clearance Area: To facilitate the smooth delivery and release of construction material ordered from outside the DRC, a customs clearance area (bonded) will be created on site from which a customs clearance official will check, register and release all imported construction material. Fenced terrace areas with a storage shed have been allowed for.

18.3 Comments on Section 18

18.3.1 Kakula Infrastructure

Infrastructure planning was completed at an appropriate level of accuracy for the Kakula 2019 PFS and no issues were identified that will have a material negative impact upon the financial viability of the project.

18.3.2 Kamoa Infrastructure

Infrastructure planning was completed at an appropriate level of accuracy for the Kamoa 2019 PFS and no issues were identified that will have a material negative impact upon the financial viability of the project.

Since the completion of the Kamoa 2017 PFS, Kansoko has been placed under care and maintenance. Some of the infrastructure, such as the 120/11 kV, 15 MVA mobile substation, has been or will be repurposed at Kakula.

The finer particle size for tailings will have an influence on water demand and more testing is needed to quantify the impact. A contingency has been included to allow for increased raw water demand.

19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies and Offtake Strategy

IDP19 assumes that the copper concentrate production will be sold at the established industry standard terms. These terms envisage a fixed concentrate treatment charge (TC) and a fixed refining charge (RC). The current long-term market outlook for TC $80/dmt concentrate and RC $0.08/lb Cu. The payable copper for concentrate can vary based on the prevailing concentrate markets but is typically 96.5% of the full copper content less a minimum deduction of 1.0 unit. This can rise to 96.75% for grades > 35% Cu and in some cases can rise to 97% for grades > 45%. The base case for IDP19 assumes a copper price of $3.10/lb and is consistent with long term estimates and pricing used in other published studies.

For the purposes of IDP19 there is potential to sell the copper concentrate to offshore smelters in Europe and Asia as well as in both Zambia and the DRC, either directly or via merchants. Logistical simplicity would favour the sale of concentrate to the DRC and Zambian smelters in the first instance but much is dependent on the smelters blend requirements and grade of concentrate which, in turn, is dependent on the smelter technologies in place.

In the DRC, CNMC is constructing a new smelter, Lualaba Copper Smelter, which is due to be commissioned in 2020. A two phase commissioning is planned with a concentrate capacity of 500 kdmt in each stage. This smelter will rely on concentrates from the DRC but the 2-stage Side Blown Furnace/Converter technology employed has specific Cu:Fe:S ratio requirements which would mean that chalcocite dominant ores like those of Kakula would require blending prior to treatment. LCS intends to mine Pyrite in the local area for this purpose.

The Zambian smelters are Konkola Copper Mines, Chambishi Copper Smelter, Mopani Copper Mines and the First Quantum Kansanshi smelter. It is estimated that the total available Zambian copper smelter capacity for Kakula copper concentrate could be approximately 100–140 kdmtpa based on current blend estimates.

In the international markets, approximately 60-65% of all copper concentrates produced are not fully integrated with a smelter or refinery owned by the same corporate entity so are sold as custom concentrates. China is the biggest importer of copper concentrates in the world and the economic analysis has assumed freight costs to this market as even local sales prices in the DRC or Zambia would probably be adjusted to reflect this.

19.2 Copper Market Overview and Dynamics

The biggest determinants of market prices can be broadly summarized as Emerging Market demand, US Housing starts, supply disruption, scrap availability and substitution. Going forward we will see significant New Economy demand from Electric Vehicles, wind and solar power.

The copper price has long since been seen as a reliable barometer of the global economy and as such is sensitive to global macro-economic developments. Asia is the dominant consumer, accounting for almost 70% share of global consumption, as China's rising middle class and increasing urbanisation rates intensify the demand for metal in construction and infrastructure projects. It is likely that, post 2020, India and ASEAN countries will drive demand growth.

Presently Asia accounts for almost 70% of global copper usage with the predominant Chinese end-use being power, at 45% of total Chinese demand. Bloomberg estimates that copper base case demand could increase by 2% pa through 2030 with a total of 12 Mt coming from New Energy and Electric Vehicles. Electric Vehicles, which use four times the quantity of copper of a conventional vehicle will account for 20% of that and any reduction in Chinese Electric Vehicle demand could be offset by higher ROW adoption rates. Interestingly, over the next decade, any decline in Chinese demand for copper is expected to be offset by ROW increases, with 70% of the cumulative growth.

The Ivanhoe marketing study suggests that longer term, on the supply side a future of declining grades and higher costs in the key developed areas of the USA and Chile is expected. However, in the short-term additional tonnage will come on to the market bringing the market closer to balance. The copper market may be undersupplied with many new projects deferred by any short-term price challenges. It is anticipated that the copper price would have to stay consistently above $3.17/lb Cu ($7,000/t Cu) to provide an incentive to new supply. It is expected that the trend of falling TCs and RCs in the past 5 years reflects a pattern of smelting capacity growing faster than new mine supply.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Studies and Issues

20.1.1 Background

The mine licence is located in a rural area with no mining or industrial activities. Extensive urbanisation, industry, and mining occur in the nearby town of Kolwezi, 25 km to the east.

Kamoa Copper SA submitted a revised Environmental Impact Study (EIS) in January 2017 and received unqualified approval in March 2017.

The Project area is characterised by scattered, undeveloped rural villages and hamlets divided between the two groupings of Mwilu and Musokantanda. A total of 32 villages fall within the mine licence area. The population in the area has been recorded as 4,311 people (Golder, 2014), indicating a population density of approximately 10 people per square kilometre. The health services in the Project area are poor. Common diseases include malaria, tuberculosis (TB) and HIV which is often associated with HIV/AIDS infections.

The climate in the area follows a distinct pattern of wet and dry seasons. Rainfall of approximately 1,225 mm is experienced annually in the region with the majority of rainfall events occurring during the period of October through to March (the wet season), with peak precipitation from December to February. The dry season occurs from April to September. The average air temperature remains very similar throughout the year, averaging approximately 22°C. At the Kamoa-Kakula Project the average annual temperatures vary between 16° and 28°C, with the average being 20.6°C. Winds at the Project are expected to originate from the east-south-east 20% of the time and south-east 14% of the time. Wind speeds are moderate to strong, with a low percentage (11.25%) of calm conditions (<1 m/s).

Topographically, the Project area is at the edge of a north–north-east to south-south-west trending ridge which is incised by numerous streams and rivers. The elevation of the Project area ranges from 1,300 to 1,540 mamsl. Current exploration activities are at elevations ranging from 1,450 to 1,540 masl. The local topography of the Project is affected by the drainage catchments of the Mukanga, Kamoa, Tshimbunji, Lufupa and Lulua Rivers and the Kalundu, Kansoko, and Kabulo Streams (Figure 20.1). Recorded water quality indicates that ground and surface water quality is generally good, well within the DRC and international (World Bank) guideline limits although natural copper concentrations exceed these limits.

Ecologically, the Project area lies within the Central Zambezian Miombo Ecoregion. This ecoregion covers a large area, stretching north-east from Angola including the south-east section of the Democratic Republic of the Congo, the northern half of Zambia, a large section of western Tanzania, southern Burundi and northern and western Malawi. The climate is tropical, with a long dry season, up to seven months, which leaves the forest vulnerable to fires, and a rainy season from October to March. The woodland is interspersed with Dambos, (grassy wetlands), which may constitute up to 30% of the region. The woodlands in the study area contain typical Miombo flora of high trees with a poorly defined shrub layer. Typically, it has more evergreen trees than most Miombo woodlands. Approximately 50%–75% of the study area is currently considered to be degraded due to agriculture and charcoal production. Sensitive habitats include shrublands, Dilungus (large flat grassland areas forming the watersheds of most of the streams and rivers in the study area. They are comprised of extensive and deep sandy soils and act as sponges retaining rainfall and releasing water into the local drainage during the dry season), Dambos (valley bottom wetland areas) and the Miombo forest to the east of the Project area (Figure 20.2).

Radiation surveys carried out indicated that radiation levels are comparable with global background levels. As the natural background levels are not elevated these do not pose an increased radiation risk to the public.

20.1.2 Summary of Environmental Studies Conducted

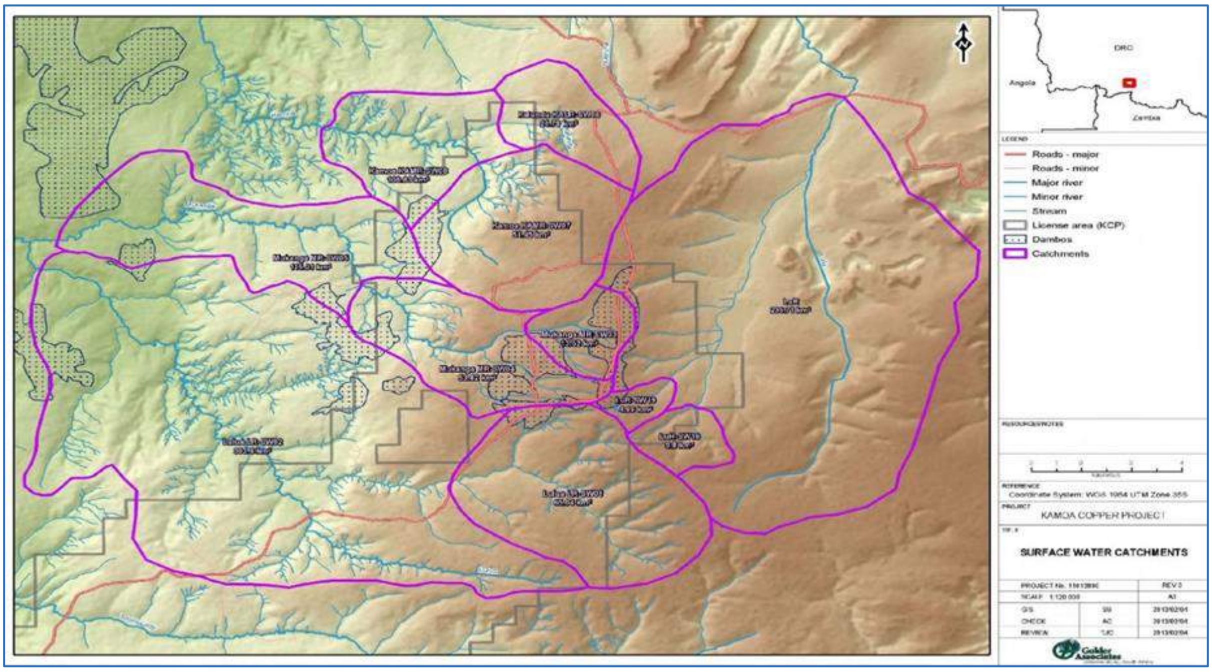
After carrying out exploration from 2006 to 2011, Kamoa Copper SA (Kamoa) (then known as African Minerals (Barbados) Limited (AMBL)) made an application to the Government to start mining in 2011. Authorisation to mine (called an exploitation licence) was given in August 2012. The application submitted by AMBL, included a description of the proposed Project (initial feasibility study) and an Environmental Impact Study (EIS) as required by DRC mining and environmental regulations, specifically - the Mining Code (Law No. 007/2002 of 11 July 2002) and the Mining Regulations, (Decree No. 038/2003 of 26 March 2003). The EIS provided an evaluation of environmental and social impacts of the Project and provided a list of actions the Project would implement to reduce the impacts and enhance or improve the benefits of the Project.

The EIS (African Mining Consultants, 2011) presented a provisional mining plan comprising of an underground copper mine for exploiting vast tonnages of high-grade ore through room-and-pillar mining, with surface processing to produce copper concentrate. Preliminary mine infrastructure locations were presented in the EIS. These included the locations of the Tailings Storage Facility (TSF) and supporting infrastructure such as employee accommodation, stores, access road and power supply. This EIS was based on conceptual planning information. This has subsequently been updated through the ongoing studies which continued since 2012.

Following the completion of the initial EIS, Kamoa Copper SA continued exploration activities and discovered additional resources. Further engineering and mining evaluation through the ongoing studies resulted in changes to the configuration of the Project. This initially involved the addition of a smelter (the addition of the smelter was subsequently deferred to Phase 2 and removed from the scope of the EIS update) and changes to the location of infrastructure, requiring an update of the EIS. It was further intended to upgrade the EIS to International Best Practices. Kamoa appointed Golder Associates (international engineering and environmental consultants (DRC registered)) in June 2012 to update the EIS, taking account of Project changes, to DRC legal requirements and to international standards. The update of the EIS, apart from updating the EIS itself, also included the following:

- Improved Project information.
- Plan of study (the Terms or Reference (ToR) to update the EIS informed by detailed scoping.
- Environmental, social and health studies (inclusive of ongoing monitoring).
- Ongoing community, interested and affected party as well as Government consultation.

Figure 20.1 Surface Water Catchments



Kamoa Copper Project Environmental Impact Study Update, Golder, 2017. Figure by Golder, 2017.

Figure 20.2 Environmental Map of the Project Area

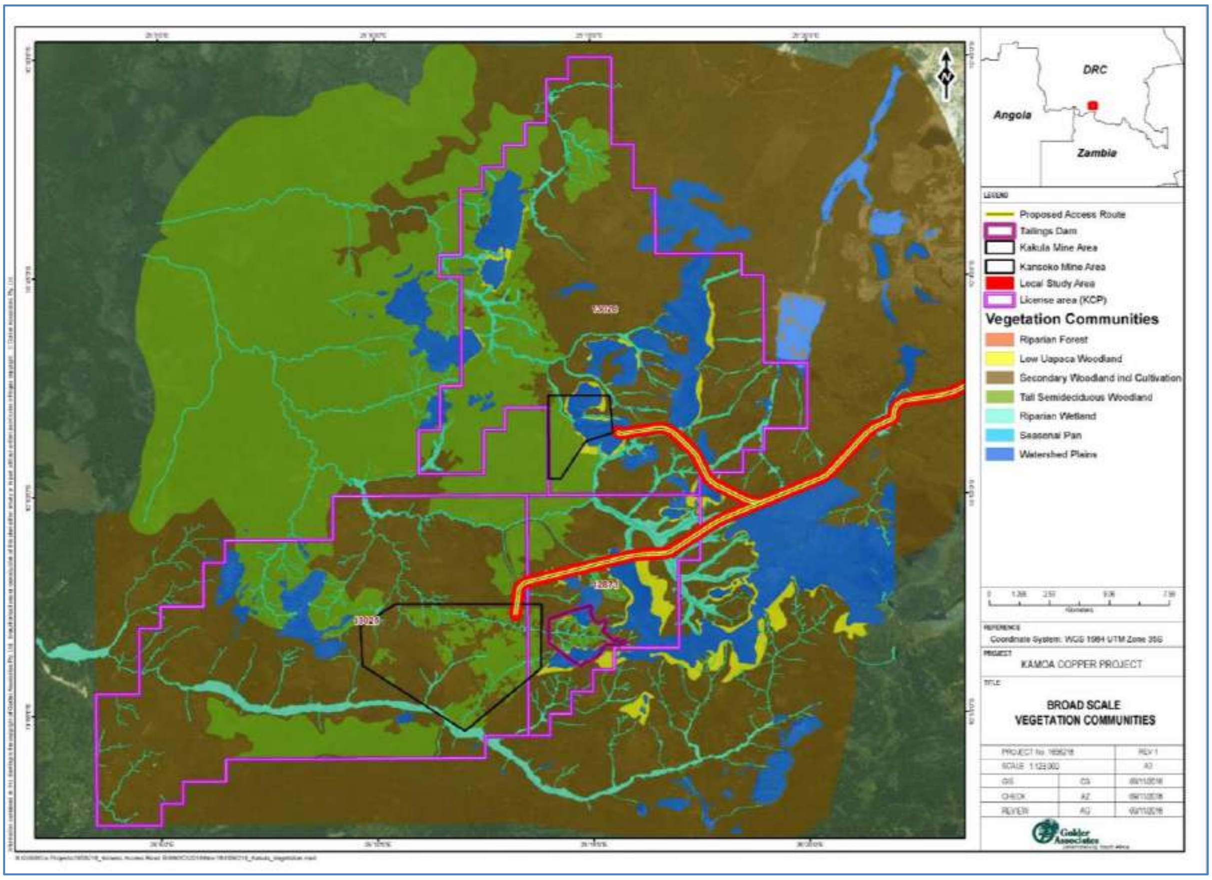


Figure by Golder, 2017.

The EIS update commenced in June 2012 with the collection of environmental, social and health data, stakeholder consultation and the development of a detailed scoping report and Terms of reference (ToR). The baseline data collection, scoping and ToR were completed in March 2014 (Golder, 2014). The work was put on hold in April 2014, pending finalisation of the project design for Phase 1, although Kamoa continued with baseline data collection. The EIS recommenced in the third quarter of 2016 with updates to the ToR, stakeholder consultation, the completion of the impact assessment and Environment and Social Management Plan (ESMP), further consultation and finalisation of the EIS update. The EIS update was submitted to government in January 2017 and approved in March 2017. The Project approved covers the two mining operations, the Kansoko Mine and the Kakula Mine. The mine development plan approved is for the production of 750,000 tonnes of copper concentrate per annum through the mining of a combined total of 8 Mt of copper sulphide ore. Government authorisation covers all the infrastructures of the two Kansoko and Kakula mines located within the three Exploitation Permit Nos. 12873, 13025, and 13026. It is noted that Kamoa commenced with initial development of a box-cut and decline in June 2014, based on the approved project description as presented in the original EIS (African Mining Consultants, 2011). The decline development was included in the EIS update. Furthermore, the Kakula box-cut and decline development was included in the approved March 2017 EIS update.

Third party developments in support of the Project include the development of the road from Kamoa-Kakula to Kolwezi airport (approved under the EIS update, 2017), and a power line from the national grid to the plant and various upgrades to existing hydroelectric schemes (approved under separate authorisation processes). In March 2014, a financing agreement was signed between Ivanhoe and the DRC's national electricity company, La Société Nationale d'Electricité (SNEL).

An Environmental Impact Study (EcoEnergie, 2013) was drafted for the power line; SNEL intends to update this EIS following project finalisation for submission to Government (an additional 12 km of the powerline servitude is required to be permitted). Regulatory environmental approvals are not required for the upgrades to the existing hydroelectric schemes.

As per the requirements of the DRC Mining Code – Kamoa generates the following additional environmental reports which are submitted to the regulator:

- Annual environmental reports.
- Bi-annual environmental third-party audits by a DRC certified environmental consulting company.
- Annual DPEM audits.

Internally the following reporting is undertaken:

- Weekly and Monthly reports; this presents a list of community incidents, grievances, stakeholder engagement, environmental incidents, environmental non-compliances, sanctions and fines and HSEC incidents.
- Weekly and Monthly monitoring reports covering – surface and ground water, dust fallout and noise.
- The KPS waste rock will be contained in a waste rock dump designed to handle the potentially acid generating pyritic siltstone.

20.1.3 Environmental Issues

Possible environmental issues that could materially affect the ability to extract the Mineral Resources or mineral reserves relating to current development/operations were determined utilising the following methodology:

- Review of Environmental and Social Reports.
- High level risk assessment of material issues utilising the following methodology:
  - Identification and listing of issues that could have an impact on Mineral Resource/Reserve extraction. These included permitting, legal non-compliance, highly sensitive environmental/social features and spatial/geographical features.
- Categorisation as follows:
  - None – issue will not impact mineral extraction.
  - Low – issue is unlikely to affect mineral extraction, would only result in disruption or delay for a short (less than one week) period of time and can easily be mitigated.
  - Medium – issue is likely to affect mineral extraction, would result in a moderate (one week to one month period) disruption or delay and can be mitigated.
  - High – issue is highly likely to affect mineral extraction, would result in extensive (>1 Month) disruption or delay to mineral extraction and cannot easily be mitigated.

From the application of the above approach, one low rated risk was identified for the current development/operation as per the initial phase of the Project:

- A number of options have been considered for the decline development to ensure that the workings remain dry, including the construction of dewatering boreholes, sealing the decline with grouting and others. The abstracted water will be discharged into the receiving environment. No environmental issues are anticipated since the discharged water reflects unaffected water quality that is well within international and DRC effluent quality guidelines. The possible surface erosion due to the discharge will be mitigated by means of dedicated energy dissipation measures.

Two low risks were identified for future developments /operations (Phase 1):

- The EIS would need to be updated and approved prior to the development of major mining related infrastructure not included in the approved mine plan. These include the TSF, process plant, workshop and stores, contractor's camp and other associated infrastructure. If the EIS is not completed and approved on time, this could delay project development.
- Resettlement and compensation of persons who may be affected by physical and economic displacement. This needs to be completed prior to infrastructure development. If the Resettlement Action Plan (RAP) is not implemented prior to construction this could delay the Project. Further resettlement requirements will be identified at the completion of the EIA modelling to determine resettlement based on impacts.

The EIS update and RAP would need to be approved and the RAP implemented prior to Project construction. Kamoa Copper SA intends to commence with the Kakula RAP studies in May 2018 and this will specifically target the tailing facility storage infrastructure.

20.2 Waste, Tailings, Monitoring and Water Management

20.2.1 Waste

Kamoa has prepared the Terms of References to commission a Waste Management Plan. This work will be integrated as part of the EIS update. The overall objectives of the Waste Management Plan are:

- To manage waste in a manner that reduces, reuses, recycles and/or recovers the majority of waste with the aim of reducing waste to landfill. A firm has been identified that can recycle used oil, discussions are underway to examine partnership options.
- To identify options available for project waste management considering the remote location of, and limited access to, power for the Project.
- To provide a cost-benefit analysis of options for waste processing activities.
- To provide a detailed integrated plan to implement waste management prior to onset of construction activities.
- To identify innovative means of waste management at Kamoa which could include one or more community managed Small Medium Enterprise (SME) project(s).
- To ensure that waste management at Kamoa is conducted in a legally compliant manner.

20.2.2 Tailings Management and Disposal

In 2014 Epoch undertook a site selection study which determined that the site termed Mupenda is the preferred site for storing the tailings from Kansoko. The Mupenda site was selected over the other options for the following reasons:

- The topography and soil properties are such that it will not require expensive measures to both contain the tailings and prevent ground water and surface water contamination.
- It is able to cater for additional storage capacity should the LOM be extended or the tailings production rate increased.
- The risks associated with this site were deemed the lowest out of the other options due to the contaminated catchment downstream of the TSF due to the historical failure of the Potopoto TSF, as well as little to no people residing downstream of the site.
- High level costs showed that this site would have the lowest costs to construct a TSF.
- The TSF will be constructed as a double valley impoundment dam with a compacted earth impoundment wall. This will have the following features:
    - The TSF impoundment walls will be constructed as a downstream facility.

  - The wall is to be raised in 7 phases, where Phase 1 is at elevation 1465 mamsl and the last phase is at elevation 1495 mamsl.
  - A final phase (Phase 8), comprising a smaller upstream impoundment wall will be constructed on top of Phase 7 when the rate of raise reduces to <1.0 m/year.
  - The TSF has a total footprint area of 540 Ha, a maximum height of 52 m and a final rate of rise of <1.0 m/year.
- A Return Water Dam with a storage capacity of approximately 45,000 m^3.
- A concrete lined Return Water Sump with a water storage capacity of 2,000 m^3.
- A slurry spigot pipeline along the crest of the TSF.

The assumptions adopted for the Kansoko TSF are the following:

- Sufficient and suitable construction materials for the preparatory earthworks associated with the TSF can be sourced from the TSF basin and nearby borrow pits.
- The legislation that has been adopted for the purpose of this study is "Appropriate Best Practice Measures". "Appropriate Best Practice Measures", in this case, implies the use of the South African Tailings Disposal Facility Design Standards and Codes (i.e. SANS 0286:1998 – "Code of Practice for Mine Tailings") amongst others.
- Furthermore, the local DRC laws regulating TSFs have been taken into account, that stipulate:
  - Appropriate measures must be taken to ensure that no toxins from any tailings storage areas enter into the groundwater. Different requirements are applicable depending on the geochemical nature and toxicity of the tailings product.
  - Surface erosion problems shall be controlled by preferably planting vegetation. Erosion problems in unconsolidated materials shall be eliminated by reducing the hydraulic gradient. If materials of different particle grading's are placed in contact with each other, appropriate filter criteria must be observed.
  - It is understood that the tailings has been classified as a "leachable mine waste" by Golder Associates (Pty) Ltd (Golder), therefore areas where the in-situ material have a permeability greater than 1x10-6cm/s must be lined with an appropriate liner system, according to the DRC Regulations. A geochemical evaluation of a tailings sample generated from a pilot plant using Kamoa ore obtained from the geological exploration boreholes was conducted (Golder, 2015). The results from the Toxicity Characteristic Leach Procedure (TCLP) showed exceedances of copper and iron in relation to Annexure XI of the DRC Mining Regulations. This was consistent with the 20-week kinetic testing results. The tailings are therefore classified as a Leachable Mine Waste that require an engineered barrier if the underlying soil does not display a permeability of ≤1x10-6 cm/s over a depth of 3 m. The majority of the TSF footprint meets this criterion, apart from the Kalahari sand which has a permeability of 1x10-5 cm/s.
  - The area underlain by Kalahari Sand is approximately 55 ha along the north-east trending Chamilundu drainage line. A suitable engineered barrier to reduce the permeability over this area to meet the criterion will have to be constructed. Provisionally, an HDPE liner has been proposed. This will be reviewed with the next phase of the Project.

20.2.3 Environment Resources

The Kamoa Copper SA environmental management team comprises eight permanently employed staff members working within the five environmental management pillars shown in Table 20.1. The team is supported by external consulting expertise as required.

Table 20.1 Environmental Management Pillars – Kamoa

Biodiversity	**Surveillance (Monitoring)**	**Waste Management**	**Performance and Compliance**	**EIS/RAP**
Reclamation, rehabilitation and restoration	Water	4 R's (Reduce, Reuse, Recycle, Recover)	Land disturbance permit	Indicators
Characterisation and biodiversity	Noise	Solid Waste	Inspections	Monitoring and evaluation
Reforestation	Air Quality	Hazardous Waste	Incidents	Impact Management
Nursery	Vibration	Liquid Waste	External Audits	Exclusion Zone
Biodiversity Action Plan	Meteorology	Mine Waste	Other – oils, petrol, diesel management etc	Livelihood
–	–	Composting	–	Risk Management

20.2.4 Site Monitoring

Environmental, social and health baseline data collection and ongoing monitoring has been carried out within the study area since 2010. In summary, this includes the following:

- Climate – Kamoa currently collates meteorological data from Kolwezi Airport located approximately 25 km east of the Project and since 2010 has been recording meteorological data at a dedicated meteorological station on site and four dedicated rain gauges. The site meteorological station will be upgraded to a fully established professional station during 2016.
- Air Quality - Kamoa undertook an ambient air quality monitoring campaign at 24 sites from April to December 2012. The pollutant parameters monitored included total suspended particulates (TSP), nitrogen dioxide (NO_2), sulphur dioxide (SO_2) and ozone (O_3). Subsequent to the initial monitoring campaigns, Kamoa Copper SA is currently undertaking dust fallout monitoring at 10 monitoring sites. The dust fallout monitoring was initiated on 06 July 2013 and was undertaken on a monthly basis until the end of 2014. Results indicate that the air quality is very good. The site currently undertakes dust fallout monitoring of 9 sites 4 times a year.

- Noise and Vibration - A baseline noise monitoring campaign was carried out by African Mining Consultants (AMC) on seven occasions between November 2010 and September 2012. Ongoing noise monitoring is undertaken by Kamoa on a weekly basis. Results indicate noise levels are below guideline limits, except near villages which are caused by human activity not related to the project. Vibration monitoring was conducted in the box-cut area and Israel village during the blasting campaigns in 2015.
- Soils, Land Use and Land Capability – three soil, land use and land capability surveys have been undertaken for the Project since 2010 covering all infrastructure locations.
- Surface Water Hydrology – a total of 14 surface water sites are currently being monitored on a quarterly basis for both quality and flow. Initially from 2010 monitoring was on a monthly basis; this was changed to quarterly once seasonal variations were understood. Results indicate good water quality conditions apart from the Luilu River downstream of the failed Potopoto Tailings Dam which has been impacted by historical mining operations.
- Groundwater – A groundwater monitoring programme has been in place since 2010. The monitoring network was expanded to incorporate many of the boreholes drilled during the 2012 and 2013 PFS drilling programme. Monthly water level monitoring is undertaken at 64 boreholes located in the Project Area. Water level loggers are installed in 7 boreholes. Sampling for water quality monitoring is undertaken quarterly at 51 boreholes. Results indicate good water quality conditions, although some areas indicate pH levels lower than the recommended WHO standards due to the natural geological formation of the area.
- Geochemistry – testwork (leach tests and kinetic tests) is being undertaken to determine Acid Rock Drainage (ARD) and Metal Leaching risk of potential waste rock, tailings and run-of-mine (ROM) ore stockpiles as per the Global ARD Guidelines with results compared to DRC regulations to determine the required mitigation measures for the waste rock dump, TSF and ore stockpiles. Results will be presented in the updated EIS.
- Radiation – A once off radiation survey was undertaken in August 2012 and included a gamma survey Soil, water, sediment and vegetation sampling and airborne dust activity sampling taken at the proposed mining area. Results indicated normal radiation conditions and limited radiological risk.
- Ecological – aquatic, terrestrial, wetland and ecosystem goods and services evaluations have been undertaken over two seasons in 2012 and over one season in 2011 and in 2016. Ecological monitoring is undertaken on an ongoing basis. Two members of the environmental department are responsible for all issues regarding biodiversity. In 2014, Kamoa put in place a nursery aimed at future rehabilitation and restoration. Progressive rehabilitation has been adopted as a practice to ensure impact minimisation and understand best practices. A plot near the Kamoa camp has been reserved for reforestation purposes and an agreement is in process with nearby communities to replicate this initiative at the community level.
- Social – Three socio economic surveys have been undertaken in the Project area: by Kamoa in 2010 and 2011 and by Golder in 2013.

- Resettlement and Compensation – Land requirements for Phase 1 will result in the economic and/or physical displacement of approximately 80 to 100 households in two villages. This is for the construction of the Mupenda TFS. Additional resettlement needs will be determined during the EIA modelling and will include impacts based on noise, dust, access, and safety.
- Economics – macro- economic data (GDP, tax, income rates and employment levels) from secondary sources was collected for the DRC and Katanga province in 2013 by Golder for EIS purposes.
- Health – specific health information was collected for the Project in 2013 by Golder for EIS purposes.
- Archaeology – archaeological and cultural heritage surveys of proposed infrastructure development areas were undertaken in 2011 for the initial EIS and 2013 for the update of the EIS by Golder.

20.2.5 Water Management

The water demand for Kamoa Copper is estimated to be 9.1 ML/d for the 6 Mtpa mine and 18.2 ML/day for the 12 Mtpa production option. Groundwater from the lower basal sandstone regional aquifer is the preferred source of bulk water supply and will be obtained by groundwater from the Southern wellfield located between 6 and 8 km to the south-west of the mine site on the southern portion of the Makalu Dome, (Golder 2015). The bulk water supply will be augmented by the dewatering boreholes to be to be drilled along the line of the decline, and could be augmented by water from the Haute Luilu Dam (Golder, 2014); this latter is considered only as a long-term contingency. Potable water for the Project will also be obtained from the wellfield.

The numerical flow modelling undertaken as part of the hydrogeological study has indicated that groundwater ingress will be relatively limited, at an average inflow of approximately 7 l/s per km^2 of mining void, (Golder, 2014). In future the bulk water supply could be augmented by excess underground mining water make as the mine void increases in spatial extent.

Studies are currently underway to determine impacts on water sources and management plans to address these will be developed as part of the updated EIS. These will include the preparation of a stormwater management plan for the entire mining complex, sized to convey the 100-year flood peak, and the development of a mine water balance. The water balance will be used to size pollution control dams to meet the one in 100-year spill frequency.

20.3 Project Permitting

As per the applicable DRC mining law and regulations, exploitation licences are mandatory before carrying out any mine activity. Kamoa was issued with the following by the DRC competent authorities for Exploitation Permits 12873, 13025, and 13026:

- On 31 January 2012, the Department for the Protection of the Mining Environmental approved the EIS (African Mining Consultants, 2011).
- The Ministry of Mines issued three Exploitation Permits to Kamoa on 21 August 2012.
- The updated EIS (2017) was approved on 3 March 2017.

20.3.1 Financial Guarantee

In terms of the financial guarantee required by DRC Law, the EIS included an estimate of the total closure costs, amounting to US$8.1 M. The DRC Mining code requires payments as financial guarantees. Kamoa Copper SA has made the payments required to the end of 2017 totalling $495,000 as per the payment schedule. The closure cost estimate and financial guarantee provision will be updated during the course of the EIS update process.

20.4 Social and Community Related Requirements and Plans

Kamoa recognises the importance of effectively doing its business by incorporating within its day-to-day management all the stakeholders concerns. These are not only its obligations to mitigate or compensate local communities for environmental and social impacts caused by the project but a community investment as added-value investment built on local identified initiatives. Nevertheless, the two are interrelated components of a holistic approach for managing company-community relationships.

Kamoa has developed a Strategic Sustainable Development Plan SSDP whose overall goal is to form the basis of and guide implementation of social and economic measures during mine construction, operation and closure that will:

- Minimise negative environmental, social and economic impacts, and maximise benefits.
- Interlink environmental, social and economic dimensions to capitalise on opportunities and benefits.
- Leave a positive legacy beyond mine closure, and thus a contribution to sustainable development.

The timeframe of the SSDP is 5 years (2016 to 2020). The scope of the current SSDP is limited to:

- Primary affected people, i.e. those living within the mining concession.
- Secondary affected people, i.e. those living in the Musokantanda and Mwilu Chiefdoms but outside the mining concession. Regional, including Mutshatsha territory and Kolwezi town.

The Specific objectives of the SSDP are to:

- Invest time, expertise and resources to provide economic opportunity, improve the quality of life and foster goodwill in the communities living around Kamoa through locally relevant initiatives.
- Engage with relevant stakeholders including local authorities, communities and their representatives, inter-governmental and non-governmental organisations and other interested parties to support projects that benefit the communities associated with Kamoa's operations.
- Assist in creating sustainable cooperatives for the benefit of local people through partnerships thus contributing to addressing food security issues and assists in the building of self-sustaining economies in the communities and improving people's standard of living.
- Assist in creating strong and reliable Small and Medium Sized Enterprises (SMEs) for local and provincial economic development through a defined enterprise support system. Develop SMEs to act as potential local suppliers of goods and services to the mining sector.
- Develop robust Livelihoods/agriculture training facility that will contribute to the skills development, productivity and economic growth.

The existing Kamoa SDP has defined its investment into four main areas:

- Stakeholder engagement.
- Livelihoods creation and improvement through the Kamoa Sustainable Livelihoods Project.
- Community economic development.
- Community skill transfer.

A community-needs assessment was undertaken by Kamoa in 2014 within communities that will potentially be affected by its operations. The needs assessment provided indications on the most important concerns for these communities and their priorities. Community projects were then proposed based on the results of the need assessment exercise.

In 2015, the Lufupa and the Luilu sectors where the Kamoa Copper SA is carrying out its operations developed their own Local Development Plan with the assistance of local NGO (SADRI and Alternative Plus) financed respectively by Cordaid and GIZ.

From the recommendations made by Investissement Durable au Katanga (IDAK), Kamoa engaged the Lufupa sector and set up a workshop to align its SDP to the Lufupa LDP. The workshop took place from 10 to12 March 2016 at Musokantanda and involved community leaders, Government representatives, two mining companies (Kamoa Copper SA and Kalongo Mining) and two NGOs (SADRI and ACIDH). Based on outcomes of this workshop the SSDP has been updated to align with the key objectives and strategies of the Lufupa LDP (Table 20.2). Further updates will be undertaken once the Lufira sector LDP has been fully developed. A total of US$1,519,000 has been budgeted for the implementation of the SSDP.

Table 20.2 Social Management Pillars – Kamoa

Community Development	**Economic and Livelihoods Development**	**Health, Safety and Human Rights**	**Stakeholder Engagement**	**ESHIA/RAP/SEMP**	**Regional Development**
Education	Agriculture	HIV	Grievance management	Indicators	Partnerships
Literacy	Small and medium sized enterprises	Malaria	Donations	Monitoring and evaluation	Technical training
Rural Infrastructure	Micro-finance	Tuberculosis	Communication	Resettlement working group	Capacity building
Water	Technical and professional training	Nutrition	Cultural Heritage	Impact management	Health
Sport	Local employment procedure	Road Safety	Stakeholder mapping	Exclusion zone	Education
Community agreements	–	Support health services	Partnerships	Livelihood restoration	FIO implementation
Capacity Building	–	Human Rights awareness	Corporate social partnership	Risk Management	–
Electricity	–	–	–	–	–

20.4.1 Social / Community Issues

Adopting the risk assessment approach outlined in Section 20.1, no social or community issues were determined to have a moderate or high risk of material impact on the ability to extract the Reserves or Resources. It should be noted that local and national electoral issues in DRC might lead to political unrest which could impact mineral extraction.

20.4.2 Risks Identified by Kamoa

Through its ongoing risk assessment and evaluation as part of its Sustainability Management System (see Section 20.5), Kamoa has identified the following key risks and management strategies (Table 20.3).

Table 20.3 Kamoa Risk Assessment, July 2015

Risk Description	Consequence	Management Strategy
High level of expectations from the population.	Frustration and unsustainable dependency.	Public disclosure during the EIS update and implementation of the Stakeholder Engagement Plan (SEP). Communication consistencies. Kamoa Copper SA policies. Local Development Plans.
Employees: Strikes, sabotage.	Reputation, relationship with government deteriorates, financial loss, project delays.	Human Resources strategy. Local hiring procedure. Union relationship, talks between management and employees.
Permitting.	Project delays and financial impacts. Relationship with Government deteriorates.	Ongoing engagement.
Deterioration of the water, air and soil quality and deforestation.	Impacts on water, soil flora and fauna. Loss of social licence. Reputational issues.	Monitoring. Erosion control. Updating of the EIS. Implementation of the Environmental and Social Management Plan (ESMP).
Employment expectations.	Community blockades and loss of social licence.	Public disclosure during the EIS update and implementation of the Stakeholder Engagement Plan (SEP). Communication consistencies. Local hiring procedure.
Influx.	Poverty, pressure on natural resources, and pressure on existing community services could result in reputational issues and loss of social licence.	Updating of the EIS. Implementation of the ESMP. Housing strategy for workers. Demography tracking, grouping of social infrastructures that incite settlement.
Resettlement: Time and inadequate resettlement due to time constraints, previous survey has resulted in expectations.	Project delays and risks associated with reputation. Lack of compliance with IFC standards.	Completion of the EIS update and RAP in good time. Approval by board. Stakeholder Engagement Plan for RAP.
Pressure from local authorities and limited capacity.	Project delays, frustration, community mobilisation, unsustainable dependency, reputational issues.	Harmonise and align the Strategic Sustainable Development Plan to the Local Development Plans.

20.5 Sustainability Management System

In 2014, Kamoa put in place a Sustainability Management System comprising of the following areas:

- Sustainability Management System - Management System procedure.
- Policy Leadership and Commitment - Sustainability policy, Environmental and Social responsibilities and accountabilities.
- Hazard Identification and Risk Management - Risk Assessments and Risk Register.
- Legal and Other Requirements - Register of legal obligations. Sustainability objectives and targets, Social and Environmental Improvement Plans.
- Objectives Targets and Performance Management – specific indicators, reporting parameters.
- Training Awareness and Competence - Induction, training and awareness material, Training Needs Analysis, training attendance registers and records.
- Communication, Consultation and Participation - Sustainability team meetings records, Stakeholders Meetings register, Stakeholder Communications and Stakeholder Engagement Plan.
- Documentation and Document Control - templates to develop documents, document control process and register, records of Approval Request Forms.
- Operational Control - Relevant documentation to manage social and environmental aspects (e.g. waste management procedure, compensation rates, H&S plan for the communities, etc.).
- Change Management.
- Emergency Preparedness and Response - Records of emergency response exercises, link to the site Emergency Response Plan.
- Contractor Management - Specifications for contractors. Incidents and grievances process and supporting templates. Incidents and grievances.
- Incident and Grievances Reporting and Management registers - Records of incidents and grievances management (e.g. investigation reports and filled grievances forms).
- Monitoring Audits and Review – Monitoring programs and outcomes.

20.6 Mine Closure

The original EIS (African Mining Consultants, 2011) presented an initial framework closure plan. This work also included the determination of the mine closure costs that were based on market knowledge, past costing and the consultant’s experience. This was then updated in 2017 for the latest EIS update.

The Kamoa 2017 PFS currently defines a 26-year mining plan; however, the resource is sufficiently large to support multiple expansion phases that could extend the life of the mine well beyond 26 years. The mine will undergo decommissioning and closure in accordance with DRC regulatory requirements at the time it is decided to close the mine.

Mine decommissioning and closure will be conducted with the following in mind:

- Creation/reinstatement of physical stable and lasting landforms.
- Protection of public health and safety.
- Limiting, and preferably obviating, predictable environmental effects, both physically and chemically.
- Reinstatement of meaningful next land use.
- Sustainability of the social programmes, including livelihoods and resettlement.
- Stakeholder engagement for closure.
- Reinstatement of meaningful land functionality.
- Optimisation of the possible social and economic benefits that could be derived from the mine in its closed state. If it is practicable, the mine will cede mine buildings, infrastructure, equipment and materials to the nearby communities to sustain/enhance local social and economic activity. This could also include the possible ongoing use of access roads created for the purpose of mining.

The key mining related infrastructure and related aspects that will require attention at mine decommissioning and closure include the following:

- Underground mine workings and related infrastructure.
- Waste Rock Dumps (WRD) and overburden spoil heaps.
- ROM pad and ROM stockpiles.
- Metallurgical Processing Facility.
- Workshops, stores and administration buildings.
- Tailings Storage Facility (TSF).
- Transport infrastructure such as site access roads, bridges and road drainage channels.
- Waste storage dams and mine site drainage systems/networks.

The decommissioning and closure of the above would in most cases follow routine practices such as removal of remaining contaminated soils and deep burying of these within the TSF before final rehabilitation, shaping and covering of outer slopes and upper surfaces of the WRD and remaining overburden piles, etc.

As underground mining methods will be followed, surface subsidence is possible. If surface subsidence occurs it should be limited and could be rectified by means of routine surface infilling, shaping and levelling.

The performance and success of the implemented closure measures will be checked and tracked by means of dedicated post closure inspection and monitoring programmes. The monitoring programmes will specifically focus on possible adverse effects on watercourses and groundwater within the zone of influence of the closed mine, reinstatement of landscape functionally (including vegetation establishment) as well as those aspects that pose potential adverse health risks and/or dangers to the public. The latter would include possible surface subsidence due to caving.

The above performance and success inspections and monitoring will be conducted by reputable independent third-party contractors. The outcomes of this work will be reflected in annual post-closure performance reports. These reports will be submitted to DPEM and made available to stakeholders as required. In those cases where the closure measures are not performing as designed, corrective action will be conducted.

The mine closure costs cover mine site decommissioning and closure measures as well as post closure inspections and monitoring as outlined above. The estimated full decommissioning and closure costs as at 2011 for the Project amount to US$8,122,375. This includes US$1,624,475 for closure management by independent third-party contractors (25% fee). The costs assume that rehabilitation and closure work is also carried out by third party contractors and that no revenue would accrue from the sale of mine equipment and/or demolition material to offset these costs.

It is noted that the current developments /operations only include the construction of a box-cut for the decline to the planned underground workings. A network of dewatering boreholes may be established to dewater the box-cut for construction to proceed. The estimated closure costs for this initial work equates to about 5% of the above estimated overall costs.

As part of the EIS update the decommissioning and closure plan and associated costs will be reviewed and updated to align with current generally accepted good practice and international standards in this regard.

21 CAPITAL AND OPERATING COSTS

21.1 Kakula 2019 PFS

Capital and operating costs for the Kakula 2019 PFS have been estimated for each of the following areas:

- Additional drilling.
- Underground mining.
- Additional power.
- Temporary facilities.
- Infrastructure.
- Concentrator.
- Indirect Costs.
- General and Administration.
- Rail.
- Transport.
- Closure.

All costs are in 2018 US$. Table 21.1 indicates the foreign exchange rates used in the estimate.

Table 21.1 Foreign Exchange Rates

Currencies	Rates ($)
ZAR/USD	14.00
CD/USD	0.73
EUR/USD	0.79

Table 21.2 summarises unit operating costs, whilst Table 21.3 provides a breakdown of operating costs on a per tonne basis. The capital costs for the project are summarised in Table 21.4.

Table 21.2 Kakula 2019 PFS Unit Operating Costs

	Payable Cu (US$/lb)		
	Years 1–5	**Years 1–10**	**LOM Average**
Mine Site	0.43	0.46	0.59
Transport	0.31	0.31	0.31
Treatment and Refining Charges	0.15	0.15	0.15
Royalties and Export Tax	0.20	0.20	0.20
Total Cash Costs	1.08	1.11	1.24

Table 21.3 Kakula 2019 PFS Operating Costs

	Total LOM US$M	**Years 1–5**	**Years 1–10**	**LOM Average**
		US$/t Milled		
Site Operating Costs				
UG Mining	4,585	34.37	35.19	38.30
Processing	1,549	12.86	12.35	12.94
Tailings	25	0.20	0.18	0.21
General and Administration	816	6.38	6.02	6.82
SNEL Discount	-212	-2.45	-2.47	-1.72
Customs Duties	347	2.68	2.70	2.90
Total	7,111	54.04	53.97	59.44

Table 21.4 Kakula 2019 PFS Capital Cost Summary

Description	Initial Capital US$M	Expansion Capital US$M	Sustaining Capital US$M	Total US$M
Mining				
Underground Mining	287	339	633	1,259
Capitalised Preproduction	107	–	–	107
Subtotal	394	339	633	1,367
Power				
Power Supply Off Site	64	–	–	64
Capitalised Power Cost	–	–	–	–
Subtotal	64	–	–	64
Concentrator and Tailings				
Process Plant	190	125	219	534
Tailings	24	15	83	122
Subtotal	214	140	303	656
Infrastructure				
Plant Infrastructure	109	124	187	419
General Infrastructure	–	–	–	–
Rail Link	–	–	–	–
Subtotal	109	124	187	419
Indirects				
EPCM	56	40	4	100
Owners Cost	103	25	–	128
Customs Duties	29	22	38	90
Closure	–	–	69	69
Subtotal	188	88	111	387
Capital Expenditure Before Contingency	968	690	1,234	2,893
Contingency	110	88	62	259
Capital Expenditure After Contingency	1,078	778	1,295	3,152

21.1.1 Underground Mining Cost Estimates

This section describes the parameters and the capital and operating cost basis of estimates to support the Kakula 2019 PFS. Unit costs are based on the most recent cost information from similar projects and adjusted where required to fit the mine plan. All costs are based on 2018 US$.

Underground Capital Costs

The total capital cost includes both preproduction and sustaining capital. Preproduction capital includes all direct and indirect mine development and construction costs prior to the start of feed through the processing plant. The cost of initial mining equipment purchased by Ivanhoe for use by the Contractor for the preproduction development is also included. After the initial development is completed by the underground Contractors, the equipment fleet used for preproduction will be used for sustaining mine development activities.

Sustaining capital is comprised of ongoing capital development and construction as well as mobile equipment rebuild and replacement costs.

The underground capital costs were estimated for the following:

- Portal.
- Underground Development – declines and primary development.
- Mobile Equipment – purchase, rebuild, and replacement.
- Fixed Equipment – including rock handling conveyors and tips.
- Surface Materials Handling Facilities with Boreholes (explosives, fuel and lube, Backfill and concrete/shotcrete).
- Initial Electrical, Control, Communications, and Instrumentation Systems.
- Main Workshops, satellite workshops with Offices and Stores.
- Underground Materials Handling Facilities (explosives, fuel and lube, concrete/shotcrete).
- Ore Bins with Feeders and Belts.
- Piping Services and Water Handling.
- Dewatering System.
- Ventilation Raises, Fans, Controls.
- Mine Air Refrigeration
- Mine Management Owners Team.
- Training of Underground Miners during the Preproduction Period.
- Contingency Mining Cost.

Underground Operating Costs

Unit operating costs were prepared for room-and-pillar stoping and drift-and-fill. Annual operating costs were generated based on the tonnes produced each year.

The underground operating costs were estimated for the following:

- Access Development for Room-and-Pillar and Drift-and-Fill.
- Production Direct Costs.
- Materials Handling Operation and Maintenance.
- Ground Support Rehabilitation.
- Dewatering.
- Ventilation and Refrigeration.
- Engineering / Mining Stores.
- Training.
- Indirect Operating Costs - not directly allocated to production.
- Power Costs.
- Undefined Allowance.

Estimate Accuracy

Stantec prepared the estimates to a Class 3 (PFS) level of accuracy (from -15% to +25%). The first two years (2019 and 2020) are detailed monthly. The following four years are detailed quarterly (2021–2024). The remaining years of mine life are detailed on an annual basis.

Sunk Costs

The estimate excludes all sunk costs up to the bottom of the north main declines (through 31 December 2018).

Contractor Profit, Overhead, and Allowances

For Contractor development and excavation, the Stantec estimate includes an 18% profit and overhead (markup) applied to equipment, materials, and labor costs. For underground Contractor construction activities, the DRA estimate applies preliminary and general (P&G) allowances to each individual activity. For the purpose of this Study, all indirect costs and P&G allowances will be referred to collectively as indirect costs.

Units of Measure

The estimate is based on the following SI units of measure.

- Meters (m) for linear distances (e.g., pipe runs, lateral development).
- Square metres (m^2) for areas (e.g., wire mesh, clearing).
- Cubic metres (m^3) for volumes (e.g., concrete, underground excavations).
- Tonnes (t) for weight (e.g. ore, waste).
- Kilograms for weight (e.g. explosives, fabricated steel).
- Liters per second (L/s) for flowrate (e.g. pumping).
- Meters cubed per second (m^3/s) for volumetric flow (e.g., ventilation).

Classifications and Cost Types

For this Study, the preproduction period is defined as starting 01 January 2019, following completion of the excavation of the north main declines, which were considered sunk costs and are not included in the estimate. Existing on-site contractor equipment is available for the Study period; therefore, capital equipment was reduced accordingly. The preproduction period is 2 years 3 months, and the ends upon mill start-up, end of Q1 2021. The production period is from the end of preproduction through the end of mine life.

Project cost estimates are broken into the following three main classifications (preproduction, sustaining capital, and operation) and two cost types (direct and indirect).

Preproduction Capital

Preproduction capital is capital costs incurred during the preproduction period. Initial capital costs are defined as all costs necessary to establish a physical asset and comprise direct and indirect costs, including the following.

- Mine surface infrastructure
- Perimeter drift development, large excavations, shafts/raises, and construction necessary to support production.
- Rock waste transportation.
- Mine fixed and mobile equipment.
- Ore stockpile costs in the portal area prior to the start of the mill or at the end of the preproduction period—whichever occurs first.
- First fills and commissioning.
- Spares for the preproduction period.
- All capital indirect expenses, including utilities and staff, required to establish the physical asset.
- Generated and line power for capital activities.

Preproduction capital does not include any sunk expenditures, like exploration, which were covered in previous Authorizations for Expenditure.

Preproduction Operating

Preproduction operating costs, defined as those incurred during the preproduction period, include the following.

- Connection drift development.
- Room-and-pillar activities.
- Drift-and-fill activities.
- Conveyor, crusher, and tip operating costs.
- Main ventilation fan operating costs.
- Main dewatering pump system operating costs.
- All indirect expenses, including facilities and staff, to support ore development and production activities.

Sustaining Capital

Sustaining capital costs include capital costs that are incurred after the preproduction period and during the production period, including the following.

- Ongoing capital development (e.g., primary drifts, large excavations, ventilation raises, additional capital facilities, additional conveyors, and tips). (This excludes connection drift development, which is considered an operating cost.)
- Additional capital equipment required to ramp up to full production.
- Annual capital required for rebuilding/replacing Owner mobile equipment that have served their designated life.

Operating

Operating costs include the non-capital costs incurred (both direct and indirect) following the preproduction period, including labor, materials, utilities, and other related costs. They include all fuel, lubricants, and all non-capital repairs.

Direct and Indirect Costs

Capital and operating costs were subdivided into direct and indirect costs, based on the Study's work breakdown structure (WBS) or the organization of the expenditure schedule.

Direct costs are cash costs directly associated with the output of a unit of production (i.e., per meter of development). These costs are within the control of the operator and immediate supervisor, including the following:

- Direct costs to produce the unit output (e.g., metres, tonnes).
- Rubber-tired ore or waste transport to a shared system or shared dump point.
- Maintenance parts, wear parts, diesel fuel, and lubricants for utilised equipment.
- Direct production and maintenance labor.
- Temporary/expendable supplies (e.g., explosives, vent tubing).
- Permanent materials (e.g., shotcrete, rock bolts).
- Rock handling.
- Direct fixed equipment operating less power (face fans and pumps).

Indirect costs are cash costs that are allocated over a group of processes and are generally not directly associated with the output of a specific unit of production. They include the following:

- Allocated Site Support.
- Recruitment.
- Training.
- General and Administration.
- Technical Services.
- Dry Facility.
- Supervision.
- General Maintenance Workshops.
- Maintenance and Mine Planning Activities.
- Central Ventilation System and Cooling System.
- Dewatering System.
- Materials Handling, Warehouses, and Laydowns.
- Electrical Power.
- Rock Handling, including Surface Stockpile.
- Spill Cleanup.
- Road Maintenance.
- Compressed Air System.
- Potable, Service, and Fire Water Supply.
- Personnel Transportation.
- Communications and Control Systems Operation.

- Health and Safety Activities.
- Sanitary Facilities Operation.
- VSAT Personnel.
- Underground Waste Handling (general garbage, tramp metal, used fluids, construction wastes, used parts and tires).

Contingency

The contingency provides additional Project capital for expenditures that are anticipated, but not defined, due to the level of engineering detail in this Study. Stantec evaluated all the non-DRA cost item groups in the expenditure schedule and applied a contingency percentage to each based upon engineering study completeness. The weighted average contingency result was 17.8%. DRA applied a 15% contingency against their estimate items.

Contractor Costs

Contractor mobilization is considered a sunk cost and was not included in the Study. Contractor demobilization is considered minimal, as the strategy is to retain equipment and facilities. Therefore, Contractor demobilization is subject to a contract that is already in place and was not considered for this Study. Contractor activities are scheduled to end in Q4 2024.

Labor

The Contractor's labor rate schedule includes the following.

- Wages.
- Overtime Allowance.
- Absentee Allowance.
- Payroll Burden.
- Work Premiums.
- Vacation Bonuses.
- Site Allowances.
  - Small Tools.
  - PPE.
  - Transport.
  - Accommodation.

Overtime was calculated at 1.3 x rate for overtime working hours exceeding 45 h/wk and 2 x rate for holidays. Kamoa Copper SA provided monthly labor rates for local and expat labor. The local rates were adjusted to a 4-day-on/4-day-off rotation, with 12-hour shifts and the appropriate overtime and holiday adjustments. The expat monthly rates were adjusted by 21.7 d/mo and 12 h/d to obtain an hourly rate for HD.

Contractor and Owner local labor rates are the same. Contractor expats were given a 42% hourly increase over Owner expats.

Permanent Materials

Permanent materials include all materials such as concrete, timber, support steel, etc., installed or consumed while performing the specified task. It is assumed that the Contractor will provide all permanent materials and supplies for their work. The general material waste factor is 5%. Some materials that, by experience, have a usage factor greater than 5% over the design quantity have a larger factor.

Direct Charge Equipment

Direct charge equipment includes specialised equipment written off by the Contractor while performing the specified task. Rental rates were not applied to this equipment since it will be either entirely written off or salvaged. Items that fall into this category include work stages, concrete forms, etc.

Equipment Operating Costs

Equipment operating costs include costs associated with operating all equipment owned or operated by the Contractor. Operating costs typically include fuel, lubrication, repair parts, overhaul parts, tire replacement, and ground-engaging components (if applicable), but excludes electrical power.

Equipment Rentals

Equipment rental costs include rental rates plus markup for Contractor-owned equipment used to complete all tasks from January 2019 to Q4 2024. Equipment rentals are charged at a monthly rate, which is 3.25% of total replacement value plus markup for this Study. The rental rate covers the contract strategy of no cost being incurred to transfer ownership to the Owner in Q4 2024.

Services and Supplies

Services and supplies include consumable items such as explosives, drilling costs, pipelines, ventilation duct, etc., associated with the specific task.

Subcontractors

Subcontractors includes subcontractor costs such as drain hole drilling, diamond drilling, assaying, etc., associated with the specific task.

Contractor Indirect Costs

Contractor indirect costs include costs incurred by the Contractor to complete specific mine development and construction activities. Contractor indirect costs were calculated on a zero-basis level.

Contractor Profit and Overhead

The Contractor's profit and overhead (markup) were assessed at 18% of the Contractor's direct and indirect costs.

Owner Costs

Permanent Capital Equipment

Permanent capital equipment includes the costs associated with purchasing fixed and mobile equipment. In addition, rebuild and replacement costs were assessed against mobile equipment. Data from other recent projects and additional Vendor supplied quotations were used to develop permanent capital equipment costs that were missing from DRA-supplied equipment costs.

To assess permanent capital costs, equipment lists were developed from infrastructure designs and operating parameters. Following are the key elements that were used to develop the unit cost database.

- Item Description – Identifies and sometimes provides a brief technical description of the equipment duty requirements or capacity.
- Base Cost – A base cost as quoted by a vendor or taken from a historical cost database, including the cost for options.
- Development Allowance – A 5% allowance to cover the cost of miscellaneous components, fuels, lubricants, and services required to commission a piece of equipment.
- Spares Allowance – A cost allowance for spare parts required on site. When provided, the cost of spares recommended by the vendor is included. A 3% allowance for critical spares was added to initial purchase costs of mobile equipment.
- Freight – Freight is included in the equipment costs. Additionally, freight is included for each movement of equipment as it pertains to rebuild and replacement. Sustaining freight is included as an indirect on the Owner's monthly costs.
- Total Unit Cost – The total unit cost is in three phases: initial, rebuild, and replacement. The total cost excludes taxes.
  - Initial purchase – Includes unit costs, options selected, critical spares, freight, and primary equipment (includes a 10% allowance toward an equipment simulator).
  - Rebuild – Includes a 60% of replacement value and freight.
  - Replacement – Includes unit costs with options and freight.

Once unit costs were developed, mobile equipment rebuild and replacement annual costs were estimated. In general, these costs are based on annual operating hours and estimates of the average life to rebuild and replace, which varies to suit the type of equipment. For the purposes of this evaluation, a rebuild and replacement schedule was developed. The equipment rebuild strategy applies only to primary equipment; the secondary fleet is subject to replacement only.

Engineering, Procurement, and Construction Management Allowance

Allowances for EPCM are included in the estimate. DRA, Golder, BBE and Stantec, each developed EPCM estimates for their portion of the Study.

The engineering component of EPCM is an allowance for detail engineering drawings and issued-for-construction drawings that are prepared for the construction of the facilities identified in the mine plan.

Procurement services costs are an allowance for the purchase of equipment, traveling to manufacturer's plants, and miscellaneous costs incurred during the purchase of both fixed and mobile equipment. Procurement services required after the production build-up period are included in the Owner's costs and are not included in this estimate.

A construction management team will be on site throughout the preproduction and production build-up periods. The construction management costs and the size of the team vary on an annual basis, depending on the amount of construction work scheduled.

Owner's Project Team

The Owner's Project Team will oversee the work performed by the Contractor and coordinated by the EPCM Contractor. This includes labor, daily expenditures, and all equipment operating costs. The current and proposed Kakula-specific labor, which would perform the duties of the Owner's Team, are included in the Kakula Indirects. All non-labor costs associated with the Owner's Team are included in the Kakula Indirects. All production activities will be performed by Owner personnel.

Electric Power Consumption

The power loads will include all underground mining loads along with surface ventilation and cooling and backfill system loads required for underground. Power consumption requirement calculations were calculated from first principles by Stantec's electrical group. These calculations are based on engineered equipment specifications with the application of demand and usage based on the mine plan. Pumping operational hours were calculated on water inflows, conveyor operating hours were calculated on tonnes per belt required, and fan operating hours are based on 24 h/d. The following criteria (and a detailed list of all designed loads) were used to develop the power usage and cost.

- Electric Motor Efficiency – 85%.
- Electric Motor Average Operating Load – 80%.
- Line Power Cost - $56.9/MWh (blended rate).
- Generated Power Cost – $0.35 KWhr (2019–2020).
- Grid Overland Power Cost – $0.057 KWhr (2021–2044).

A detailed annual power load sheet was prepared and annual power usage estimated based on the yearly production and estimated horsepower.

Afridex Blasting Costs – DRC

Afridex blasting costs are included in the Owner indirects. Zero-based estimating determined the appropriate quantities. The per-blast unit costs are currently under negotiation with the appropriate stake holders. Afridex blasting costs include the following:

- Afridex – $20.00/blast.
- DRC State – $58.33/blast.
- Afridex Percentage Costs – 3% of Blasting Supplies Costs.
- Blasting Licenses –$1,500/person on an Annual Basis.
- Blasting Certificates – $300/person on an Annual Basis.

Mobile Equipment Rebuild and Replacement

Mobile equipment rebuild and replacement costs are charged in the time period the mobile equipment reaches the applicable operating hours. The following method was used for the cost of mobile equipment rebuild and replacement:

- Rebuild life is variable based on the type of equipment.
- Rebuild cost is assessed at 60% of the base unit cost plus freight.
- Replacement cost is assessed at 100% of the base unit cost plus options and freight.
- There are no replacement costs in the final 2 years of development for development equipment and no replacement charges for other equipment in the final 2 years of mine operations.

Operating Cost Estimate Basis

Operating costs include the non-capital costs incurred (both direct and indirect) during the production period, including labor, materials, utilities, and other related costs. They include all fuel, lubricants, and all non-capital repairs to keep the Project functioning.

Operating Cost – Undefined Allowance

An allowance was included as 5% of the operating costs for miscellaneous costs that may not be accounted for elsewhere in the estimate. This allowance is only applied to direct production costs.

21.1.2 Concentrator and Site Infrastructure Costs

This section describes the basis and methodology used to prepare the Capital Cost Estimate (CCE) for the Kakula concentrator together with site infrastructure capital costs, project indirect costs, infrastructure cost and contingencies for the initial capital requirements for the execution of the Kakula project phase.

The Kakula concentrator design is based on a phased approach of two 3 Mtpa processing modules, as dictated by the mining ramp-up and production profile. A phased approach further allows for increased processing flexibility and plant redundancy while also reducing the peak capital demand by phasing of capital expenditure.

The scope of the CCE is restricted to the PFS scope as discussed in Section 17 and Section 18.

General

The Kakula CCE meets the required accuracy criteria of -15% +25% and complies with a Class 1 Pre-Feasibility study as defined by the DRA Estimating Study Class Matrix. Note that this is equivalent to a Class 3 Estimate as defined by the American Association of Cost Engineers (AACE). The estimate has been presented in Dollar (USD) currency in present day (November 2018) terms.

The following inputs and documents were used in compiling the estimate:

- Process flow diagrams.
- Mechanical equipment list.
- Electrical motor list.
- Site plot plans.
- General arrangement drawings.
- Electrical cable schedules and HT single line diagrams.
- Equipment Quotations from Vendors.
- Project Execution Programme.

Capital costs have been estimated for the following disciplines:

- Earthworks.
- Civil works.
- Structural steel fabrication, supply and erection.
- Platework fabrication, supply and erection.
- Mechanical equipment supply and installation.
- Pipework fabrication, supply and erection.
- Electrical, control and instrumentation (E, C&I) supply and erection.
- Infrastructure buildings.
- Transportation to site.
- EPCM services.
- First fills of consumables.
- Spares.

Bulk Earthworks and Infrastructure

The large bulk earthworks and general infrastructure services were measured from detailed drawings that were prepared for the construction phase of the project. For all other items of earthworks and infrastructure, relevant to the PFS, a high-level bill of quantity (BOQ) was produced and populated with rates as received from RLB Pentad.

The Golder's geotechnical report was further used to create typical cut and fill layer works design which was used in tendered bill of quantities (BOQs).

Geotechnical

Because of the limited geotechnical information available, the following assumptions were made:

- No other ground improvements have been allowed for except for the terrace works.
- All the overhaul distance has been assumed as 5 km.
- A ground bearing pressure of 150 kPa was assumed.
- No allowance has been made for material to be crushed and screened as this will be sourced from existing earthworks contractor established on site and operating a crushing and screening plant.
- The excavated material will be spoiled no further than 2 km from works.

Conveyor Earthworks

No bulk earthworks and terracing are required for conveyors, only restricted earthworks for civil bases and sleepers.

Access Roads and Parking

A preliminary bulk BOQ was prepared for all roads and parking. The following allowances were made for the main access road:

- Area of the road to be Cleared and Grubbed.
- Remove Topsoil.
- Cut to Spoil of 300 mm.
- Intermediate Excavation of 15%.
- Hard Rock Excavation of 10%.
- Roadbed Preparation of 300 mm Deep.
- Borrow to Fill G7 Material 300 mm Thick.
- G6 and G5 Commercial Base Layer.
- G2 Layer and Asphalt Capping.

The BOQ was then populated with rates received from RLB Pentad.

Buried Services and Storm Water Reticulation

A preliminary BOQ was prepared for all buried services piping and storm water reticulation quantified by DRA infrastructure from layout drawings. The BOQ was then populated with rates received from RLB Pentad.

Fencing

All fencing was quantified from layout drawings. A "bulk" BOQ was prepared and populated with rates received from RLB Pentad.

Pollution Control Dam

The following assumptions were made for estimating purposes:

- Pollution control dam would be excavated into terrace or platform.
- An allowance has been made for fill, for the breadth of the dam.
- No other ground improvements have been allowed for, except for the terrace works.
- The excavated material will be spoiled no further than 2 km from works.

Civil Works

The civil works requirement was quantified using the block and plot plans in conjunction with general arrangement drawings for each area. A preliminary BOQ was produced in order to detail all the items of civil works relevant to the PFS. The following general assumptions were made:

- All concrete is assumed to be 25 MPa.
- All exposed surfaces to be wood floated.
- The rebar to concrete ratio varies between 85 kg/m^3 to 110 kg/m^3.
- The BOQ was then populated with rates received from RLB Pentad.
- The civil works quantities for the following items were based on “as built” costs from previously executed projects.
- Mill feed stockpile tunnel and roof structure.
- ROM stockpile feed conveyor “Giraffe” structure.
- ROM stockpile tunnel civils.
- Rougher and cleaners flotation civils.
- Reagents store civils and structures.
- Mill building civils, structures and platework.
- Tailings thickeners civils, structures and platework.
- Concentrate thickener civils, structures and platework.

Infrastructure Building Works

Infrastructure buildings were quantified by RLB Pentad. An all-in rate per m² (benchmarked from current projects) was applied to the quantities and this cost has been included in the estimate.

The infrastructure building works P&G percentages varied between 38% to 40%.

Structural Steelwork and Platework Supply and Erection

Steelwork quantities for all structures were estimated from general arrangement and layout drawings, while platework and lining items were quantified from the equipment list, PFDs, general arrangement and layout drawings.

A preliminary BOQ was produced which included all steelwork and ancillary bought outs including sheeting, grating, stairtreads and handrailing, as well as platework and lining rates. In order to obtain current market related rates and P&Gs, RLB Pentad issued enquiries to several SMP fabrication and construction companies familiar with projects of this size and nature in the DRC. Due to the recent uncertainty of steel prices, these rates were averaged and applied to the estimate.

Rates for steel fabrication, delivery and erection were categorised as either light, medium or heavy steelwork, and were further priced per tonne of steel. The SMP rate includes for shop detailing, corrosion protection, supply, transportation to site, off-loading and erection including all their associated costs w.r.t. induction, medicals, accommodation and meals, PPE, PPC, travelling, etc. to complete the works in full.

In addition to the above, corrosion protection has been allowed for in accordance with the Kamoa Corrosion Protection Specification. The rates for surface preparation, galvanising, priming and painting have been included in the estimate as part of the ex-works fabrication and supply rate. An allowance has been provided for the touch-up of steelwork and platework on site after installation. The corrosion protection P&G costs have been included in the steelwork and platework P&Gs. Structural steel P&Gs have been taken as 51% percentage of the total supply and installation cost.

Mechanical Equipment

Mechanical equipment datasheets for all major equipment was approved by the client prior to enquiry. Enquiries were then issued to reputable vendors for costing of the following major equipment packages:

- Surface cone crushers.
- Feeders and screens.
- Ball mills.
- Regrind mills (costing received as part of a technology trade-off conducted).
- Cyclones.
- Flotation Cells and low pressure air blowers.
- Slurry and froth pumps.
- Thickeners.
- Concentrate filters.
- Bulk bagging system.

The erection cost for the mechanical equipment was based on the average SMP erection rates received from RLB Pentad. The mechanical P&Gs were applied as 51% of the erection component only.

Preliminary designs of all belt conveyors was carried out by the DRA engineering department in accordance with the belt profiles as depicted on the general arrangement drawings, including the duty to meet the process requirements and designed to meet the general engineering design criteria. Calculations for each conveyor were carried out and the mechanical equipment components and steelwork content were quantified.

Conveyor mechanical equipment was costed using a combination of budget quotes from reputable vendors and the DRA data base of recent projects.

Rates for conveyor steelwork and platework were as per the averaged SMP rates received from RLB Pentad as described in the steelwork and platework sections above. These rates have been used to populate the estimate.

Conveyor P&Gs have been taken as 51% of the erection component.

For minor and/or ancillary mechanical equipment items, supply costs were obtained from previous quotations and/or the DRA database.

Piping and Valves

All surface infrastructure buried piping as well as overland piping has been measured from layouts and plot plans and included in the civil works BOQ.

The cost of in-plant process piping and valves was derived as a 20% factor of the mechanical equipment supply cost, in line with plants of a similar size and nature from the DRA database.

Electrical, Control and Instrumentation

Electrical loads (and typical starter panel types) were assigned to all transformers and MCCs for all areas based on the mechanical equipment list. The following basis was used for the highlighted items below:

- An average LV cable run of 100 m per motor were included while the cable price was based on a enquiry issued to reputable vendors.
- An average HV cable run of 50 m between switchgear and transformer were included, while the cable price was based on a enquiry issued to reputable vendors.
- Cable racking quantities were based on per m^2 using cable width + 30% and lengths as above.
- Field Isolators, MCCs, starters, transformer costs, and costs associated with Power Factor Correction (PFC) were based on pricing from the DRA database.
- 11 kV switchgear was costed with the aid of the single line diagram and the costs were based on costs received during the execution works project.
- 220 kV switchgear was costed with the aid of the single line diagram, while the costs were based on pricing from the DRA database.
- Lighting and Small Power: No detailed engineering was completed to support the costs. Costing was based on a per square meter basis for each area.
- Mini substation was assigned to grouped loads based on the block and plot plans, while the price was based on a enquiry issued to reputable vendors.
- Pricing for low voltage distribution boards were based on the DRA database.
- Pricing for 11 kV overhead Lines were based on the DRA database.
- Conveyor belt lighting and earthing quantities were measured from the general arrangement and layout drawings, while the costs were based on the DRA database.

- Control & Instrumentation: Take offs were done to produce a preliminary BOQ from the MEL, PFDs and block plan and layout drawings. DRA database costs were applied to BOQs and the estimate was then populated with these costs. Benchmarked factors from similar previous DRA plants were used to check the control and instrumentation ratio to electrical costs. The factors were found to be in line with the benchmark plants.

The electrical, control and instrumentation P&Gs were costed by the E,C&I installation contractor from the various BOQs.

Transportation

Transportation costs for steelwork and platework were quoted in the SMP rates. Other transport costs were based on a tonnage or "per load" basis. Rates from the logistic service provider (Nucleus Mining Logistics) were used in the CCE. These rates include for all costs involved in the transportation of goods to site, including the logistics and documentation, as well as the actual transportation of the goods.

Turnkey Packages

Allowances has been made for the following turnkey packages:

- Fire detection and suppression systems.
- Fuel and lubrication strorage and distribution system.
- Sewerage treatment plant.
- Potable water treatment plant.
- Waste water treatment plant.
- Laboratory equipment.

Spares

Spare parts costs have been included in the estimate to cover commissioning, strategic and two-year operational spares for mechanical and electrical equipment. The spares holding costs have been derived from vendor recommendations, as per quotations received, and have been grouped per item of equipment in a separate section within the estimate.

Where no spares were quoted by the vendor, a percentage of the supply price has been applied as follows:

Commissioning Spares

- Mechanical equipment: 2.5% of mechanical supply.
- Conveyor mechanical equipment: 2.5% of conveyor mechanical supply.
- Turnkey packages: 2.5% of turnkey package supply.

Strategic / Capital Spares

- Mechanical equipment: 7.5% of mechanical supply.
- Conveyor mechanical equipment: 7.5% of conveyor mechanical supply.
- Turnkey packages: 7.5% of turnkey package supply.
- E,C&I: 5.0% of E,C&I supply.
- Piping and Valves: 3.0% of piping & valves supply.

Two year Operational Spares

- Mechanical equipment: 7.5% of mechanical supply.
- Conveyor mechanical equipment: 7.5% of conveyor mechanical supply.
- Turnkey packages: 7.5% of turnkey package supply.

First Fill of Consumables

Allowances have been made for the first fill of oil and lubrication, reagents and grinding media.

Preproduction Costs

Allowances have been made in the CCE for the following during the construction period:

- Construction power,
- Construction water,
- Construction laydown area,
- Construction offices,
- Construction communication,
- Construction vehicles,
- Construction SHEQ,
- Construction signage,
- Construction ablution facilities,
- Construction it and computer equipment,
- Construction access and security facilities,
- Construction waste facilities, and
- Commissioning tools.

21.1.3 Owner's Cost and G&A

Ivanhoe have prepared a budget for Owners costs. The costs include allowance for the following items:

- Office and General Expenses.
- Maintenance.
- Equipment and Sundry.
- Fuels and Utilities.
- Other Offices.
- Insurance and Insurance Taxes.
- IT Hardware and Software.
- Personnel Transport.
- Training.
- Communications.
- Licences and Land Fees.
- Labour Expatriate.
- Labour Congolese.
- Accommodation and Messing.
- Medical Support.
- Expatriate Flights.
- Light Vehicles.
- Environmental.
- Community Development.
- Banking and Audit Fees.
- Legal and Consultants.
- Studies.
- Resettlement.
- Capitalised General and Administration costs.

21.2 Kamoa 2019 PFS

The Kamoa 2019 PFS is an update of the work previously called the Kamoa 2017 PFS. The mine design, production, process plant and infrastructure have remained the same. Costs were updated by applying a total of 2% escalation to the costs.

21.2.1 Summary

Capital and operating costs have been estimated for each of the following areas: additional drilling; underground mining; additional power; temporary facilities; infrastructure; concentrator; indirect costs; general and administration; rail; transport; and closure.

The Kamoa 2019 PFS is an update of the work previously called the Kamoa 2017 PFS. The mine design, production, process plant and infrastructure have remained the same. Costs and price assumptions have been updated by applying a total of 2% escalation to the costs.

Table 21.5 summarises unit operating costs, whilst Table 21.6 provides a breakdown of operating costs on a per tonne basis. The capital costs for the project are summarised in Table 21.7.

Table 21.5 Kamoa 2019 PFS Unit Operating Costs

	Payable Cu (US$/lb)		
	Years 1–5	**Years 1–10**	**LOM Average**
Mine Site	0.64	0.59	0.66
Transport	0.51	0.51	0.51
Treatment and Refining Charges	0.18	0.19	0.19
Royalties and Export Tax	0.26	0.26	0.26
Total Cash Costs Before Credits	**1.59**	**1.55**	**1.61**

Table 21.6 Kamoa 2019 PFS Site Operating Costs

	Total LOM US$M	**Years 1–5**	**Years 1–10**	**LOM Average**
		US$/t Milled		
UG Mining	3,534	31.55	28.74	28.23
Processing	1,443	11.53	11.53	11.53
Tailings	29	0.31	0.24	0.24
General and Administration	785	7.20	6.06	6.27
SNEL Discount	-231	-2.18	-2.12	-1.80
Customs Duties	281	2.46	2.29	2.25
Total	**5,842**	**50.86**	**46.74**	**46.71**

Table 21.7 Kamoa 2019 PFS Capital Cost Summary

Capital Costs (US$M)	**Initial Capital US$M**	**Expansion Capital US$M**	**Sustaining Capital US$M**	**Total US$M**
Mining				
Underground Mining	317	–	822	1,139
Capitalised Preproduction	13	–	–	13
Subtotal	**331**	**–**	**822**	**1,152**
Power and Smelter				
Power Supply Off Site	64	–	–	64
Subtotal	**64**	**–**	**–**	**64**
Concentrator and Tailings				
Process Plant	149	85	171	405
Tailings	21	97	–	118
Subtotal	**170**	**182**	**171**	**523**
Infrastructure				
Plant Infrastructure	113	–	85	197
Subtotal	**113**	**–**	**85**	**197**
Indirects				
EPCM	66	27	22	115
Owners Cost	80	27	9	116
Customs Duties	27	8	40	74
Closure	–	–	70	70
Subtotal	**173**	**62**	**140**	**376**
Capital Expenditure Before Contingency	**851**	**245**	**1,217**	**2,312**
Contingency	144	55	142	341
Capital Expenditure After Contingency	**994**	**299**	**1,359**	**2,653**

21.2.2 Underground Mining Cost Estimates

This section describes the parameters and the capital and operating cost basis of estimates to support the Kamoa 2019 PFS. Unit costs are based on the most recent cost information from similar projects and adjusted where required to fit the mine plan. All costs are based on 2018 US$.

For Underground Mining, costs were estimated at a prefeasibility study level of accuracy, with unit costs based on the most recent cost information from similar projects and adjusted where required to fit the mine plan.

Underground Capital Costs

The total capital cost includes both preproduction and sustaining capital. Preproduction capital includes all direct and indirect mine development and construction costs prior to the start of feed through the processing plant. The cost of initial mining equipment purchased by Ivanhoe for use by the Contractor for the preproduction development is also included. After the initial development is completed by the underground Contractors, the equipment fleet used for preproduction will be used for sustaining mine development activities.

Sustaining capital is comprised of ongoing capital development and construction as well as mobile equipment rebuild and replacement costs.

The underground capital costs were estimated for the following:

- Portal.
- Underground Development – declines and primary development.
- Mobile Equipment – purchase, rebuild, and replacement.
- Fixed Equipment – including rock handling conveyors and tips.
- Surface Materials Handling Facilities with Boreholes (explosives, fuel and lube, concrete/shotcrete).
- Initial Electrical, Control, Communications, and Instrumentation Systems.
- Main Workshop with Offices and Stores.
- Underground Materials Handling Facilities (explosives, fuel and lube, concrete/shotcrete).
- Ore Bins with Feeders and Belts.
- Piping Services and Water Handling.
- Dewatering System.
- Ventilation Raises, Fans, Controls.
- Mine Air Refrigeration
- Mine Management Owners Team.
- Training of Underground Miners during the Preproduction Period.
- Contingency Mining Cost.

Underground Operating Costs

Unit operating costs were prepared for room-and-pillar stoping and controlled convergence room-and-pillar stoping. Annual operating costs were generated based on the tonnes produced each year.

The underground operating costs were estimated for the following:

- Access Development for Controlled Convergence Room-and-Pillar.
- Production Direct Costs.
- Materials Handling Operation and Maintenance.
- Ground Support Rehabilitation.
- Dewatering.
- Ventilation and Refrigeration.
- Engineering / Mining Stores.
- Training.
- Indirect Operating Costs - not directly allocated to production.
- Power Costs.
- Undefined Allowance.

21.2.3 Concentrator Costs

The capital and operational costs for the concentrator were prepared for the Kamoa 2019 PFS and are described below.

Construction costs for the process plant have been developed based on using predominantly Chinese Contractors. The viability of utilising Chinese labour will be investigated during the next phase of this project.

21.2.3.1 Concentrator Capital Cost Estimation Basis

Capital costs are defined as the expenditure required during the design, construction and commissioning phases of the project. This includes all costs associated with labour, construction, plant and equipment, bulk materials, other materials, permanent equipment, sub-contracts, packaging, transportation, loading, off-loading, strategic spares and capital indirect costs which contribute to the physical construction of the project.

Estimating - General

The following inputs and documents were identified and used in compiling the estimate:

- Process design basis.
- Site plot plans.
- Block flow diagrams.
- Process flow diagrams.
- Mechanical equipment list.
- Battery limits as described in the study documentation.

Costs have been estimated for the following disciplines:

- Earthworks.
- Civil works.
- Structural steel fabrication, supply and erection.
- Platework fabrication, supply and erection.
- Mechanical equipment supply.
- Mechanical equipment installation.
- Pipework fabrication, supply and erection.
- Electrical and C&I supply and erection.
- Transportation to site.
- EPCM services.
- First fills and spares.
- Infrastructure buildings.

Major Mechanical Equipment

Short-form enquiries were prepared and issued to three vendors for all major mechanical equipment. This category represented more than 90% of the total process plant mechanical equipment supply costs and included the following:

- Crushers.
- Feeders and screens.
- Conveyors.
- Ball mills and relining equipment.
- Regrind mills.
- Cyclones.
- Flotation Cells and Blowers.
- Slurry and Froth pumps.
- Thickeners.
- Concentrate filters.
- Bulk bagging system.

The installation costs for mechanical equipment were factorised from the supply costs and allowances were made for vendor installation, supervision and commissioning as appropriate.

Earthworks and Roads

Limited bulk earthworks have been allowed for as part of the civil bulk quantity estimate.

Surface Facilities and External Infrastructure

Infrastructure and Plant estimate quantities were obtained from models and Bills of Quantities in 76% of the cases. Rates were obtained via budget quotes from vendors in 52% of the cases.

The following surface facilities are included in the cost estimate:

- Plant water services (including plant raw, potable, gland service and process water).
- Air services (including blower and compressed air).
- Plant pipe racks.

Furthermore, the following facilities are included in the overall estimate, within the plant infrastructure category:

- Tailings Storage Facility (TSF) pipeline (19,600 m for 2 lines).
- Return water pipeline (10,400 m).
- Borehole supply pipeline (10,450 m).
- In-plant roads.
- Plant vehicles.
- Sewerage treatment plant.
- Fencing.
- Infrastructure buildings.
- Substations.
- Camps.
- Dams.
- Temporary and backup power.
- Road/rail infrastructure.

Structural Steelwork

Structural steel material take-offs were developed from layout drawings. Rates for steelwork supply and fabrication were taken from database rates.

Civil Works

Civil bulk quantities were developed from layout drawings of the plant areas. Rates for civil works were taken from database rates.

P&G costs have been quantified as part of the civil summary.

Piping, and Valves

The process plant piping and valves cost estimates were factorised as a percentage of the mechanical supply cost.

Electrical, Control and Instrumentation

Budget quotations were obtained for major electrical equipment.

The pricing of all other electrical, control and instrumentation item costs were factorised from mechanical equipment costs.

The installation cost for the quoted major electrical equipment was obtained from vendors where vendor installation is required or based on database rates from similar projects.

Transportation

Load estimates and shipping and transport budget quotes for delivery to site were based on in-house data.

EPCM

EPCM costs were built up from first principles, based on the project execution schedule and estimated based on current personnel rates.

Spares and Consumables

Costs were included in the estimate to cover operating, strategic and commissioning spares for the mechanical and electrical equipment. Allowances were made for first fills.

21.2.3.2 Concentrator Operating Costs

The operating cost estimate includes the fixed (labour and maintenance) costs and variable costs components (reagents, grinding media and power costs). The operating costs are expressed in United States Dollar (USD) per tonne milled. The operating cost figure excludes rehabilitation, mining, insurance costs, import duties and all other taxes.

The sources of information and assumptions are as follows:

- Vendor information and quotations.
- Plant labour rates and staffing levels as supplied by Kamoa Copper SA.
- Power cost ($0.0569/kWh) supplied by Kamoa Copper SA.

Crushing and Grinding Consumables

The consumables for the crushing and grinding sections include screen panels, crusher liners, mill liners and grinding media. The liner wear rate and steel ball consumption rate are estimated using the Bond abrasion index. The regrind mill ceramic media consumption is based on a vendor supplied rate which is referenced to the regrinding power consumed.

Flotation Consumables

The main flotation consumables are reagents and the consumption rates are based on the testwork performed by XPS. Reagent prices are provided by vendors.

Thickening Consumables

Flocculant consumption rates are assumed as no vendor settling testwork on concentrates or tailings has been conducted.

Power

Power consumption is based on operating power estimates of the equipment in the MEL and using estimated operating time for that equipment.

Water

As water is supplied by bores or pumping from underground workings, the cost of water is the cost of the power required to deliver it to the plant. These costs are in the power estimates.

Maintenance

A simple 5% factor has been applied to the overall mechanical equipment cost to provide a Kamoa 2017 PFS level maintenance cost estimate.

Transport for Consumables

Transport costs for delivering reagents and grinding media to site have been provided by the vendors.

Labour

The labour cost estimate is based on the labour rates and personnel numbers provided by Kamoa Copper SA.

The labour structure assumes a strong day shift presence in the plant when the bulk of the maintenance as well as all reagents off loading and make-up activities will be completed.

On-Site Laboratory

The on-site laboratory is to be operated by SGS under contract and, as such, the rates have been supplied by SGS.

21.2.4 Kamoa Tailings Storage Facility

21.2.4.1 TSF Capital Cost

The capital costs associated with the TSF have been estimated by Epoch to an accuracy of +/-25% and have been based on contractor rates. Epoch has provided the following qualifications to their estimate:

- P&G accounting for 20% of the total works. Based on DRC experience, this value may be as high as 30%–40%.
- No allowance for escalation has been made.
- The above costs exclude provisions for:
  - Pumps.
  - Mechanical and electrical and instrumentation components.
  - Pump stations.
  - Slurry and return water pipelines between the TSF and the Plant.

Closure costs have been assumed to occur after production ceases. However, some closure costs may be incurred earlier if there are opportunities for progressive TSF rehabilitation.

21.2.4.2 TSF Operating Cost Estimate

The operating costs associated with the TSF have been estimated with allowances for the following:

- Tailings deposition and operations management.
- General works associated with the TSF.
- Consulting services.

Aftercare and Maintenance Requirements

On completion of the final rehabilitation and closure works, an aftercare and maintenance program will be enacted to ensure that the closure measures are robust, have performed adequately and that no further liabilities arise. The aftercare period is normally not less than 5 years but can be extend depending on the physical and chemical characteristics of the facility. The aftercare and maintenance program allowed for is assumed to include:

- Periodic inspection of the cover and vegetation for signs of erosion damage and failures of the vegetation establishment process;
- Repairs and amendments to the closure works as necessary;
- Re-planting of areas of vegetation where required;
- Periodic inspection and monitoring to confirm the effectiveness of the closure works in achieving the stated closure objectives, including:
  - Collection and analysis of ground and surface water samples;

- Measuring of phreatic surfaces within the TSF and assessment of the overall structural stability of the facility; and
- Inspections of stormwater decant facilities for signs of damage.

No allowance has been made for the treatment of water that will need to be discharged into the environment from the TSF after closure as treatment is assumed to be unnecessary.

Aftercare costs were estimated for a period of two years following mine closure. This cost is indicative and has been based on closure cost estimates undertaken for similar operations, by Epoch.

Bulk Water Supply Capital and Operating Costs - Kamoa Wellfield

The following assumptions are made with regard to pumps, pipelines and associated infrastructure:

- The Mine will provide electrical power at each pump station.
- Limited civil work will be required to install the pumps at all the required places.
- Submersible pumps will be acceptable to be used.
- Stainless Steel pumps will only be required at the Pollution Control Dams (PCDs).
- Pipes can be laid on top of the ground.
- Joining of Pipes can be done with continuous welding.

Wellfield Development Capital and Operating Costs

A number of production boreholes are required to supply the estimated 9.1 ML/d for the Kamoa 2017 PFS production scenario.

Borefield capital is expensed ahead of production as start-up capital, while the operating, maintenance and energy costs will be incurred commencing in production Year 1.

Stormwater Management Plan

The following assumptions were made when developing the cost estimates for the stormwater management plan:

- South African construction rates were escalated 20%.
- Petrol and labour are included in the rates.
- No allowances for other escalations.
- P&G allowance of 15%.
- Engineering rates were assumed to be 12.5% of the total capital costs.
- The soil characteristics are assumed to be suitable for construction of the PCD walls and the berms alongside each channel.

Work not allowed for in the schedule of quantities and rates include:

- Box and key cut quantities and costs. The depths and configuration can only be finalised during the detailed and construction phases of the project.
- Hard excavation and blasting.
- All electrical, instrumentation and power supply items.
- All taxes (in country taxes, etc.).
- Costs for detailed design, tender documentation, code of practice, operation manual, quality assurance.
- Costs for any additional studies.

21.2.5 Power Infrastructure Rehabilitation and Upgrade

The costs of the power plants rehabilitation have been estimated by Stucky Ltd (Stucky) in its power study and updated by Kamoa Copper SA in 2015.

These estimated costs are based on equipment suited to the region.

Based on the June 2011 Memorandum of Understanding (MOU) with SNEL, the capital cost of the rehabilitation will be financed by Ivanhoe through a loan to SNEL. The loan including interest will be repaid by SNEL through a deduction from Ivanhoe's monthly power bills incurred over the life of the mine. For the financial analysis this has been assumed to be a 40% discount to the power charges and results in the discount being applied for 14 years from commencement of production.

21.2.6 Concentrate Transport Operating Costs

The base case logistics solution for the Kamoa 2019 PFS is the use the road and rail corridor between southern DRC and Durban in South Africa, which is viewed as the most attractive and reliable export route currently available. As soon as the railroad between Kolwezi and Dilolo, a town near the DRC Angolan border, is rehabilitated, there is an alternative option to transport concentrate by rail to the port of Lobito in Angola. The total costs including all fees and charges for the southern transport route were estimated to be US$349/t and for the western route via Lobito to be US$254/t.

21.2.7 Closure Costs

An allowance has been made for Closure costs in the financial model. This equates to 10% of all capital expenditure excluding Mining, Power and Indirect costs.

21.2.8 Owner's Cost and G&A

General and Administraion costs were estimated in detail for he Kamoa 2017 PFS and were escalated by 2% for the Kamoa 2019 PFS.

21.3 Comments on Section 21

21.3.1 Kakula 2019 PFS

In the QP's opinion, the work completed for the Kakula 2019 PFS adequately supports a PFS study.

21.3.2 Kamoa 2019 PFS

In the QPs' opinion, the work completed adequately supports this level of study estimate.

22 ECONOMIC ANALYSIS

22.1 Economic Assumptions

The following model and taxation assumptions are used in the Kakula 2019 PFS and the Kamoa 2019 PFS. On 9 March 2018, Law No. 18/001 amending the 2002 Mining Code was promulgated. Kamoa Copper already adopted voluntarily several measures since the entry into force of the new legal framework governing mining activities in March 2018. For the purpose of this report, the economic analysis is based on the 2018 Mining Code. Kamoa Copper will continue to monitor the regulatory provisions to be adopted, ensuring as far as possible, continued adequate enforcement of the relevant legislative requirements. Detailed discussions are ongoing with the aim of resolving, in a fair and equitable manner, the mining industry's concerns with the 2018 Mining Code.

22.1.1.1 Pricing and Discount Rate Assumptions

The Project level valuation model begins on 1 December 2019. It is presented in 2018 constant dollars; cash flows are assumed to occur evenly during each year and a mid-year discounting approach is taken.

The copper price used for the evaluation is US$3.10/lb copper. This is considered to be reasonable based on industry forecasts and prices used in other studies. The product being sold is copper concentrate and payment terms for the copper assume that the LOM average payable copper concentrate is 97.73%.

The copper concentrate assumes an $80 per tonne treatment charge and refining charge of US$0.08/lb copper. The copper concentrate transport charge (including provincial road taxes and duties but excluding the provincial concentrate export tax and DRC export tax) to the customer is assumed to be US$349/t via road to Ndola and rail to Durban for shipping for the first two years of production and thereafter US$254/t via rail through Lobito.

22.1.1.2 Taxation

The DRC Mining Code provides for all the taxes, charges, royalties, and other fees. Kamoa Copper SA provided the assumptions for taxes and royalties applicable to an operating mine in the DRC. Only material taxes that would have an impact on the financial model have been considered and require confirmation.

In the analysis, carry balances such as tax and working capital calculations are based on nominal dollars and outputs are then deflated for use in the integrated cash flow calculation. The working capital assumptions for receivables, payables are 6 weeks and 6 weeks. These assumptions are preliminary and will need to be verified in later studies.

22.1.1.3 Royalties

A company holding a mining exploitation licence is subject to mining royalties. The royalty is due upon the sale of the product and is calculated at 3.5% of the gross commercial value of non-ferrous metals 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.

Gross commercial value is determined by a coefficient depending on the nature of the product, which is 95% of total value for blister copper (91-98% Cu content) and 65% for copper concentrate (31-60% Cu content).

The holder of the exploitation licence will 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.

Pursuant to the 2018 Mining Code, a minimum contribution of 0.3% of turnover must be made to community development initiatives.

22.1.1.4 Key Taxes

The DRC Mining Code provides for all the taxes, charges, royalties, and other fees. The key taxes are listed below.

General Corporate Taxation

Companies that are the holders of mining rights are subject to tax at 30% on net income and withholding tax on distributions are subject to 10% tax at the shareholder's level. In addition, as from 01 January 2014, the minimum amount of tax payable by mining companies in a year is 1% of the calculated revenue for that specific year ("Minimum Tax Amount").

Tax Holidays

The DRC tax legislation does not currently provide for any tax holiday incentives.

Tax Losses

The aggregate exploration expenditure may be claimed.

Depreciation

Specific mining assets dedicated to mining operations, with useful lives between 4 and 20 years are depreciated as follows:

- First year: 60% depreciated based on the cost of the asset.
- For subsequent years: a declining balance depreciation is applied based on the tax years remaining over the life of the mine.

Non-mining assets are depreciated in accordance with the common law. The common low provides different depreciation rates for various assets, e.g. 10 years for plant and equipment.

VAT

VAT came into effect in the DRC in January 2012. VAT is levied on all supplies of goods and services at a rate of 16% and is not levied on any capital asset movements.

Customs/Import Duties

Customs duty will be applied separately to capital (5%) and operating costs (10%) for direct cost line.

Export Taxes

National Export Tax

The fee is limited to 1% of the value of the export.

Provincial Export Tax on Concentrate

A provincial tax on the export of concentrate is levied on a per tonne basis and equates to US$100/t concentrate exported.

Provincial Export Road and Infrastructures Renovation Tax

A provincial export tax levied on any product exported by road is also levied on a per tonne basis at a rate of US$50/t. Copper concentrate will be exported by road to neighbouring countries and will thus be subject to the Road tax.

Withholding Taxes

A Withholding tax at the rate of 14% on services supplied by foreign companies established offshore to onshore companies applies. Mining companies are liable for movable property withholding tax at a rate of 10% in respect of dividends and other distributions paid. Non-mining companies are subject to withholding tax of 20%.

Dividend Distributions/Interest Repayments

Any dividend distributions made to Ivanhoe, as well as the DRC government will attract a withholding tax of 10%. A withholding tax of 20% applies if the loan is denominated in local DRC currency. If the loan is however denominated in foreign currency no withholding tax is payable. Interest payments to any local intermediate and holding companies attract a withholding tax of 20%.

Exceptional Tax on Expatriates

In the DRC, an employer is liable for the exceptional tax on expatriate's remuneration at a rate of 25%. Mining companies are subject to 10%. It is determined in terms of the salaries generated by the work carried out in the DRC and is deductible for purposes of calculating the income tax payable.

22.2 Kakula 2019 PFS Overview and Results

Ivanhoe is developing twin declines at the Kakula Mine once in production, one will be a service decline for the transport of personnel and materials into the mine, and the second will be a conveyor decline for rock handling and transport of personnel and materials out of the mine. The base case described in the Kakula 2019 PFS is the construction and operation of an underground mine, concentrator processing facilities, and associated infrastructure. The Kakula 2019 PFS production is planned to be an average of 6 Mtpa ore over a production period of 25 years.

The Kakula 2019 PFS represents the initial phase of the Kakula development. The Kakula 2019 PFS evaluates the development of a 6 Mtpa underground mine and surface processing complex at the Kakula Deposit—a discovery announced in early 2016—as the project's first phase of development. The development scenario of the Kakula Mine on the Kakula Deposit is shown in Figure 22.1.

The Kakula 2019 PFS has an average annual production rate of 291,000 tonnes of copper at a mine site cash cost of US$0.46/lb copper and total cash cost of US$1.11/lb copper for the first ten years of operations. The preproduction capital cost of US$1.1 billion for this option would result in an after-tax net present value at an 8% discount rate (NPV8%) of US$5.4 billion.

Figure 22.1 Kakula 2019 PFS 6 Mtpa Development Scenario

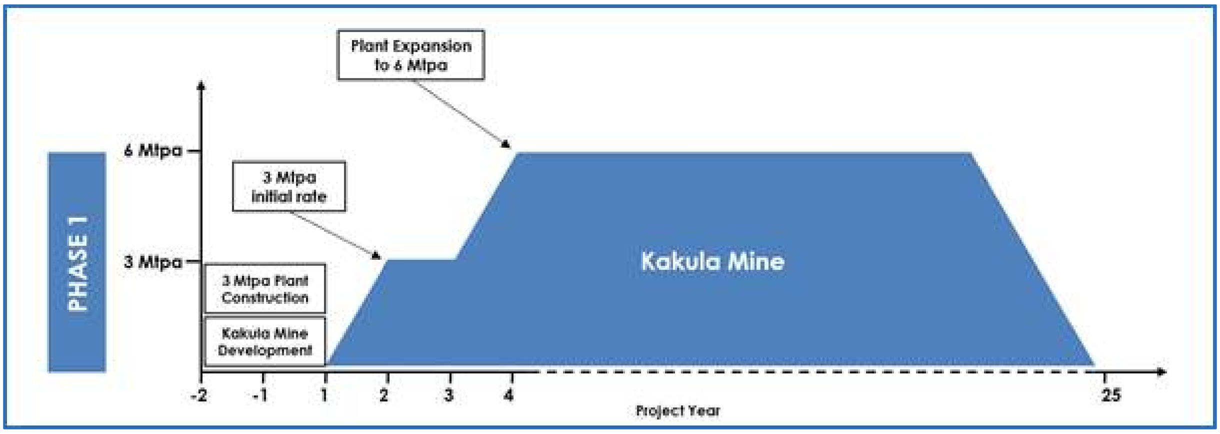


Figure by OreWin, 2019.

A summary of the key results for the Kakula 2019 PFS scenario are:

- Very-high-grade initial phase of production is projected to have a grade of 7.1% copper in Year 2 and an average grade of 6.4% copper over the initial 10 years of operations, resulting in estimated average annual copper production of 224,000 tonnes.

- Initial capital cost, including contingency, is estimated at US$1.1 billion.
- Average total cash cost of US$1.11/lb of copper during the first 10 years.
- After-tax NPV, at an 8% discount rate, of US$5.4 billion.
- After-tax internal rate of return (IRR) of 46.93%, and a payback period of 2.6 years.
- Kakula is expected to produce a very-high-grade copper concentrate in excess of 50% copper, with extremely low arsenic levels.

The Kakula 2019 PFS the Kakula mill would be constructed in two smaller phases of 3 Mtpa each as the mining operations ramp-up to full production of 6 Mtpa. The LOM production scenario provides for 119.7 Mt to be mined at an average grade of 5.48% copper, producing 9.8 Mt of high-grade copper concentrate, containing approximately 12.3 billion pounds of copper.

The economic analysis uses a long-term price assumption of US$3.10/lb of copper and returns an after-tax NPV at an 8% discount rate of US$5.4 billion. It has an after-tax IRR of 46.93% and a payback period of 2.6 years.

The estimated initial capital cost, including contingency, is US$1.1 billion. The capital expenditure for off-site power, which is included in the initial capital cost, includes a US$64M advance payment to the DRC state-owned electricity company, SNEL, to upgrade two hydropower plants (Koni and Mwadingusha) to provide the Kamoa-Kakula Project with access to clean electricity for its planned operations. Mwadingusha is being upgraded first. The work is being led by Stucky Ltd., of Switzerland; the advance payment will be recovered through a reduction in the power tariff.

The Kakula 2019 PFS describes the initial phase of the Kakula development and presents the first Mineral Reserve for the Kakula Deposit. The Kakula 2019 PFS evaluate the development of a 6 Mtpa underground mine and surface processing complex at the Kakula Deposit. The development scenario of the Kakula Mine on the Kakula Deposit is shown in Figure 1.8.

The Kakula 2019 PFS analyses the potential development of an initial 6 Mtpa Kakula Mine at the Kakula Deposit in the southerly portion of the Kamoa-Kakula Project's discovery area. For this option, the PFS envisages an average annual production rate of 291,000 tonnes of copper at a mine site cash cost of US$0.46/lb copper and total cash cost of US$1.11/lb copper for the first ten years of operations. The preproduction capital cost of US$1.1 billion for this option would result in an after-tax net present value at an 8% discount rate (NPV8%) of US$5.4 billion.

The key results of the Study are summarised in Table 22.1.

Table 22.1 Kakula 2019 PFS Results Summary

Item	Unit	Total
Total Processed		
Quantity Milled	kt	119,728
Copper Feed Grade	%	5.48
Total Concentrate Produced		
Copper Concentrate Produced	kt (dry)	9,776
Copper Concentrate	kt (dry)	9,776
Copper Recovery	%	85.35
Copper Concentrate Grade	%	57.32
Contained Copper in Concentrate	Mlb	12,354
Contained Copper in Concentrate	kt	5,604
Peak Annual Recovered Copper Production	kt	360
Ten Year Average		
Copper Concentrate Produced	kt (dry)	508
Contained Copper in Concentrate	kt	291
Mine-Site Cash Cost	US$/lb Payable Cu	0.46
Total Cash Cost	US$/lb Payable Cu	1.11
Key Financial Results		
Peak Funding	US$M	1,099
Initial Capital Costs	US$M	1,078
Expansion Capital Costs	US$M	778
Sustaining Capital Cost	US$M	1,295
Mine Site Cash Cost	US$/lb Payable Cu	0.59
Total Cash Costs After Credits	US$/lb Payable Cu	1.24
Site Operating Costs	US$/t Milled	59.44
After-Tax NPV8%	US$M	5,440
After-Tax IRR	%	46.9
Project Payback Period	Years	2.6
Initial Project Life	Years	25

Table 22.2 summarises the financial results, whilst Table 22.3 summarises mine production, processing, concentrate, and metal production statistics.

Table 22.2 Kakula 2019 PFS Financial Results

Net Present Value (US$M)	Discount Rate	Before Taxation	After Taxation
	Undiscounted	19,317	13,575
	4.0%	12,053	8,411
	6.0%	9,693	6,733
	8.0%	**7,875**	**5,440**
	10.0%	6,457	4,432
	12.0%	5,336	3,635
Internal Rate of Return		55.6%	46.9%
Project Payback Period (Years)		2.4	2.6

Table 22.3 Kakula 2019 PFS Production and Processing

Item	Unit	Total LOM	Years 1–5	Years 1–10	LOM Average
Total Processed					
Quantity Milled	kt	119,728	4,589	5,321	4,789
Copper Feed Grade	%	5.48	6.79	6.39	5.48
Total Concentrate Produced					
Copper Concentrate Produced	kt (dry)	9,776	465	508	391
Copper Concentrate	kt (dry)	9,776	465	508	391
Copper Recovery	%	85.35	85.58	85.64	85.35
Copper Concentrate Grade	%	57.32	57.32	57.32	57.32
Contained Copper in Concentrate					
Copper	Mlb	12,354	588	642	494
Copper	kt	5,604	267	291	224
Payable Copper in Concentrate					
Copper	Mlb	12,074	575	628	483
Copper	kt	5,477	261	285	219
Payable Copper					
Copper	Mlb	12,074	575	628	483
Copper	kt	5,477	261	285	219

The Kakula 2019 PFS 6 Mtpa mill feed and copper grade profile for the LOM are shown in Figure 22.2 and the concentrate and metal production for the LOM are shown in Figure 22.3.

Figure 22.2 Kakula 2019 PFS Process Production

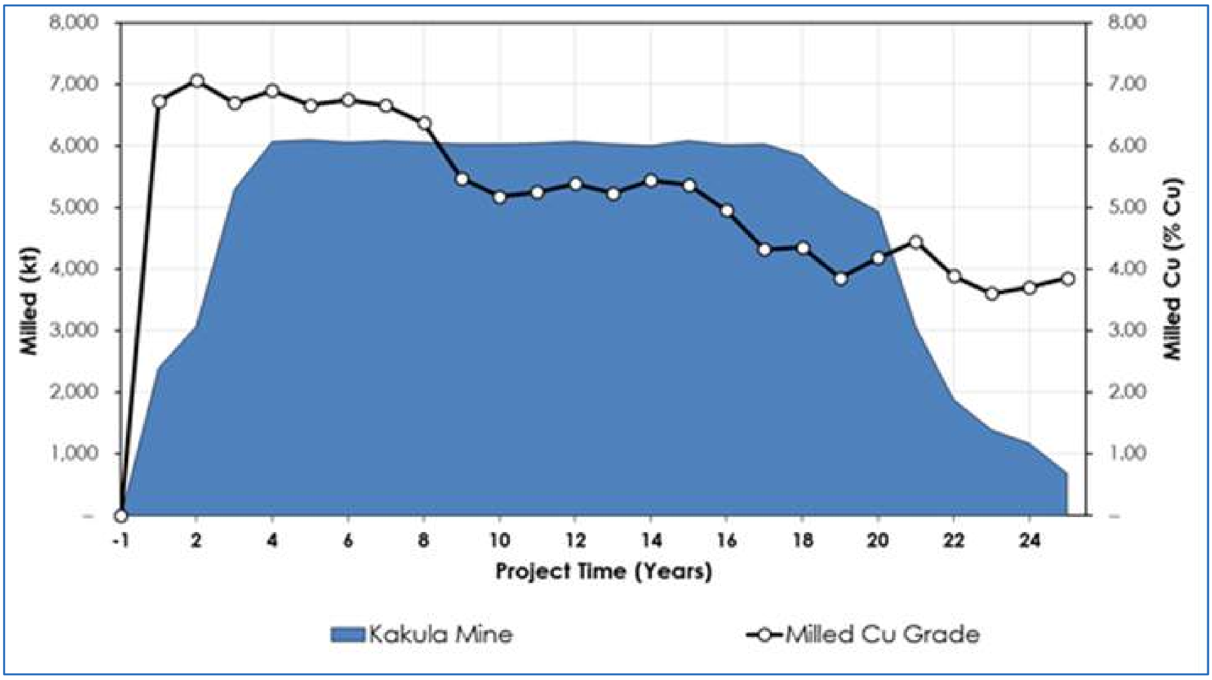


Figure by OreWin, 2019.

Figure 22.3 Kakula 2019 PFS Concentrate and Metal Production

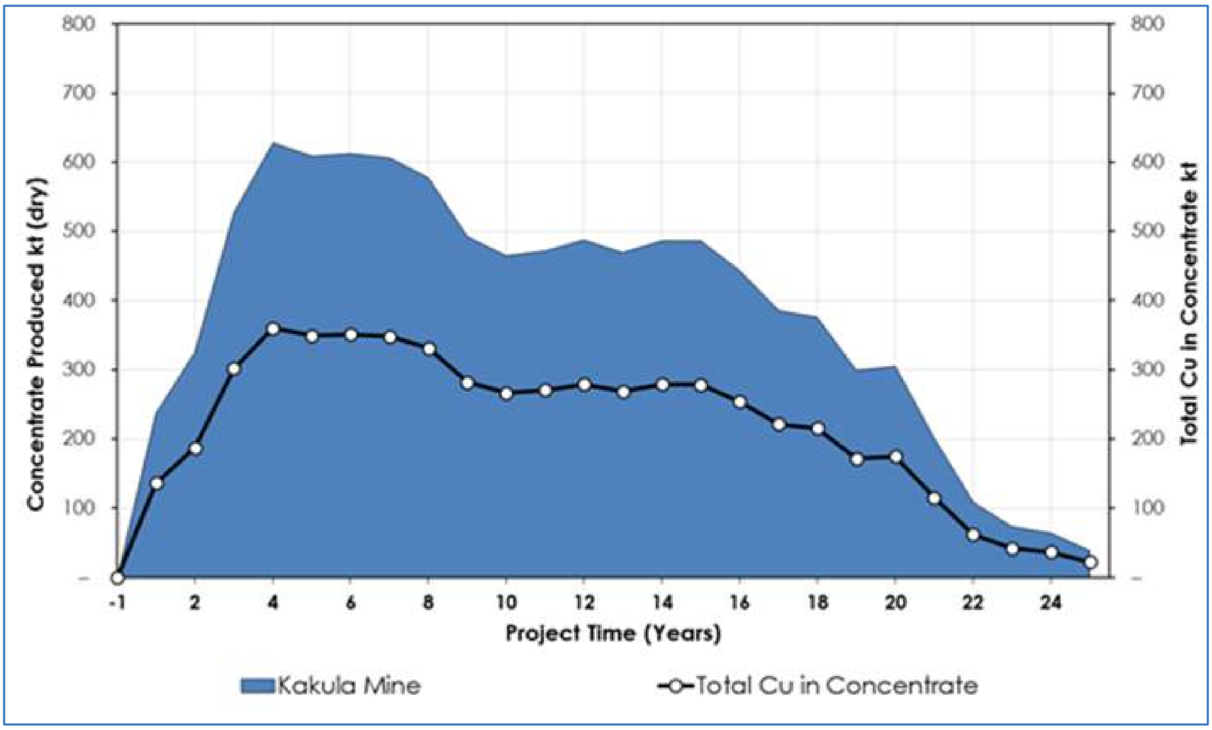


Figure by OreWin, 2019.

Table 22.4 summarises unit operating costs. Table 22.5 provides a breakdown of revenue and operating costs. Capital costs for the project are detailed in Table 22.6.

Table 22.4 Kakula 2019 PFS Unit Operating Costs

	Payable Cu (US$/lb)		
	Years 1–5	**Years 1–10**	**LOM Average**
Mine Site	0.43	0.46	0.59
Smelter	–	–	–
Transport	0.31	0.31	0.31
Treatment and Refining Charges	0.15	0.15	0.15
Royalties and Export Tax	0.20	0.20	0.20
Total Cash Costs	**1.08**	**1.11**	**1.24**

Table 22.5 Kakula 2019 PFS Revenue and Operating Costs

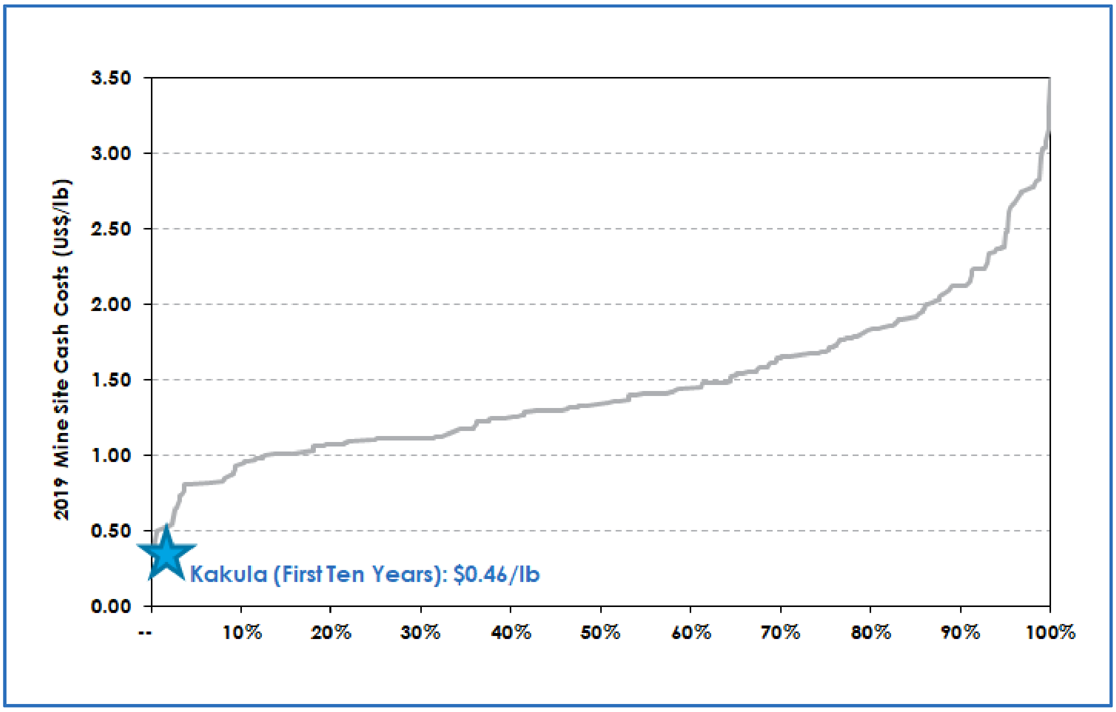
	Total LOM (US$M)	Years 1–5	Years 1–10	LOM Average
		US$/t Milled		
Revenue				
Copper in Concentrate	37,429	388.32	365.60	312.62
Gross Sales Revenue	**37,429**	**388.32**	**365.60**	**312.62**
Less: Realisation Costs				
Transport	3,707	38.46	36.21	30.96
Treatment and Refining	1,770	18.37	17.29	14.79
Royalties and Export Tax	2,403	24.93	23.47	20.07
Total Realisation Costs	7,880	81.76	76.97	65.82
Net Sales Revenue	**29,549**	**306.57**	**288.63**	**246.80**
Site Operating Costs				
Underground Mining	4,585	34.37	35.19	38.30
Processing	1,549	12.86	12.35	12.94
Tailings	25	0.20	0.18	0.21
General and Administration	816	6.38	6.02	6.82
SNEL Discount	-212	-2.45	-2.47	-1.72
Customs Duties	347	2.68	2.70	2.90
Total	**7,111**	**54.04**	**53.97**	**59.44**
Net Operating Margin	**22,438**	**252.53**	**234.66**	**187.36**
Net Operating Margin	**75.94%**	**82.37%**	**81.30%**	**75.92%**

Figure 24.5 shows the average mine-site cash cost during the first 10 years of the Kakula 2019 PFS on Wood Mackenzie's industry cost curve. This figure represents mine-site cash costs that reflect the direct cash costs of producing paid concentrate or cathode incorporating mining, processing and mine-site G&A costs.

Figure 24.6 shows the C1 pro-rata copper cash costs of the Kakula 2019 PFS on Wood Mackenzie's industry cost curve. This figure represents C1 pro-rata cash costs that reflect the direct cash costs of producing paid copper incorporating mining, processing, mine-site G&A and offsite realization costs, having made appropriate allowance for the costs associated with the co-product revenue streams.

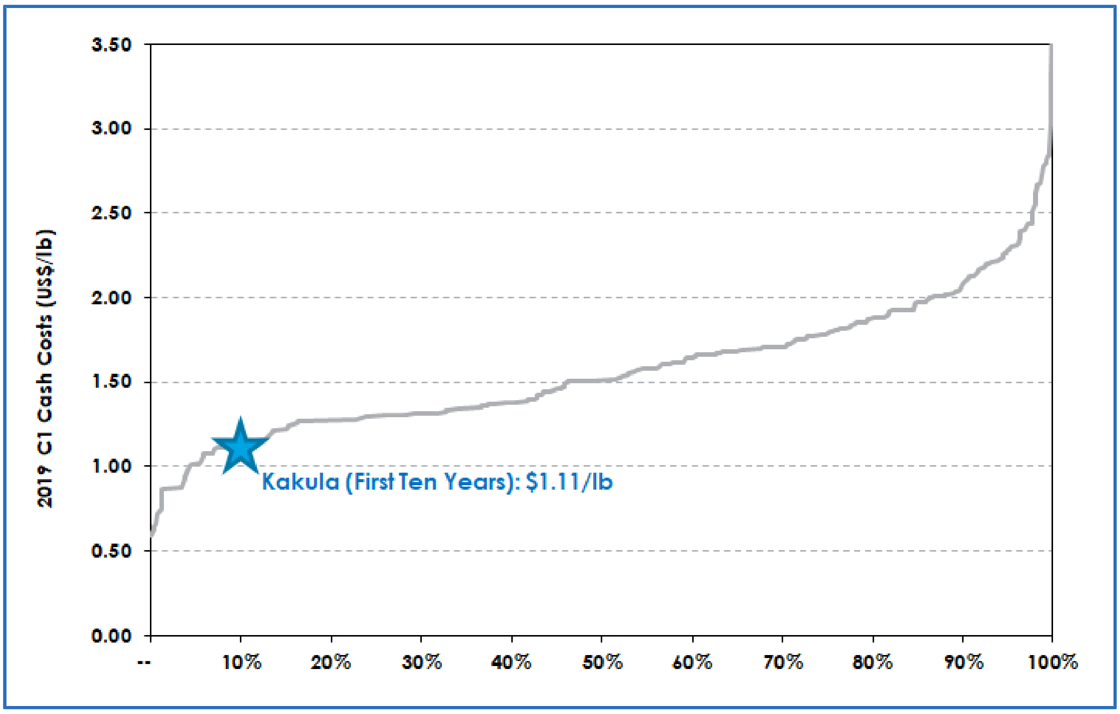
For both charts, the Kamoa Kakula IDP19 was not reviewed by Wood Mackenzie prior to filing, and information was sourced the data from public disclosures.

Figure 22.4 2019 Mine-Site Cash Costs (Includes All Operational Costs at Mine Site)



Source: Wood Mackenzie 2019.

Figure 22.5 2019 C1 Copper Cash Costs



Source: Wood Mackenzie 2019.

Table 22.6 Kakula 2019 PFS Capital Costs

Description	Initial Capital US$M	Expansion Capital US$M	Sustaining Capital US$M	Total US$M
Mining				
Underground Mining	287	339	633	1,259
Capitalised Preproduction	107	–	–	107
Subtotal	394	339	633	1,367
Power				
Power Supply Off Site	64	–	–	64
Subtotal	64	–	–	64
Concentrator and Tailings				
Process Plant	190	125	219	534
Tailings	24	15	83	122
Subtotal	214	140	303	656
Infrastructure				
Plant Infrastructure	109	124	187	419
General Infrastructure	–	–	–	–
Rail Link	–	–	–	–
Subtotal	109	124	187	419
Indirects				
EPCM	56	40	4	100
Owners Cost	103	25	–	128
Customs Duties	29	22	38	90
Closure	–	–	69	69
Subtotal	188	88	111	387
Capital Expenditure Before Contingency	968	690	1,234	2,893
Contingency	110	88	62	259
Capital Expenditure After Contingency	1,078	778	1,295	3,152

Figure 22.6 compares the capital intensity for large-scale copper projects. The figure shows projects identified by Wood Mackenzie as recently approved, probable or possible projects reported with nominal copper production capacity in excess of 200 ktpa (based on public disclosure and information gathered in the process of routine research by Wood Mackenzie). The estimates are based on public disclosure and information gathered by Wood Mackenzie. The Kamoa Kakula IDP19 was not reviewed by Wood Mackenzie prior to filing.

Figure 22.6 Capital Intensity for Large-Scale Copper Projects

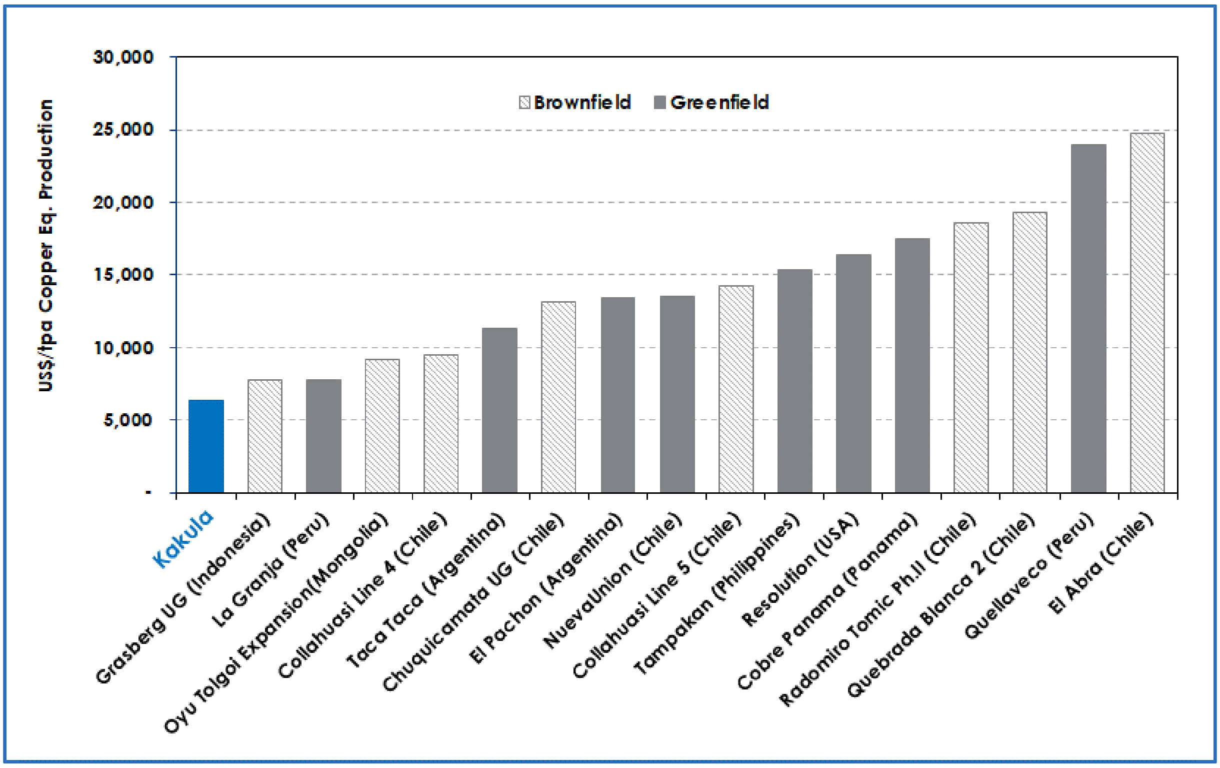


Figure by Ivanhoe, 2019. Source: Wood Mackenzie.

The after-tax NPV sensitivity to metal price variation is shown in Table 22.7 for copper prices from US$2.00/lb to US$4.00/lb. Cost sensitivity is shown in Table 22.8. The annual and cumulative cash flows are shown in Figure 22.7 (annual cash flow is shown on the left vertical axis and cumulative cash flow on the right axis).

Table 22.7 Kakula Mine Copper Price Sensitivity

After Tax NPV (US$M)	Copper Price - US$/lb					
Discount Rate	**2.00**	**2.50**	**3.00**	**3.10**	**3.50**	**4.00**
Undiscounted	4,541	8,656	12,757	13,575	16,840	20,920
4.0%	2,694	5,298	7,892	8,411	10,478	13,061
6.0%	2,070	4,194	6,310	6,733	8,418	10,525
8.0%	1,581	3,339	5,090	**5,440**	6,835	8,577
10.0%	1,195	2,670	4,138	4,432	5,600	7,060
12.0%	887	2,140	3,386	3,635	4,626	5,863
15.0%	535	1,535	2,529	2,727	3,516	4,501
IRR	22.7%	34.7%	45.0%	46.9%	54.1%	62.5%

Table 22.8 Kakula Mine Cost Sensitivity

			Change from Base $NPV_8\%$ (US$M)				
Variable	**Units**	**Base Value**	**-25%**	**-10%**	**–**	**10%**	**25%**
Initial Capital Cost	US$M	1,078	5,609	5,508	**5,440**	5,372	5,270
Expansion Capital Cost	US$M	778	5,557	5,487	**5,440**	5,393	5,323
Initial and Expansion Capital Cost	US$M	1,856	5,726	5,555	**5,440**	5,325	5,152
Site Operating Cost	US$/t Milled	59	5,964	5,650	**5,440**	5,229	4,910
Treatment and Refining	US$/t and US$/lb Cu	80/0.08/110	5,572	5,493	**5,440**	5,388	5,309
Transport	US$/t Conc	349/254	5,718	5,552	**5,440**	5,329	5,162

Figure 22.7 Kakula Mine Projected Cumulative Cash Flow

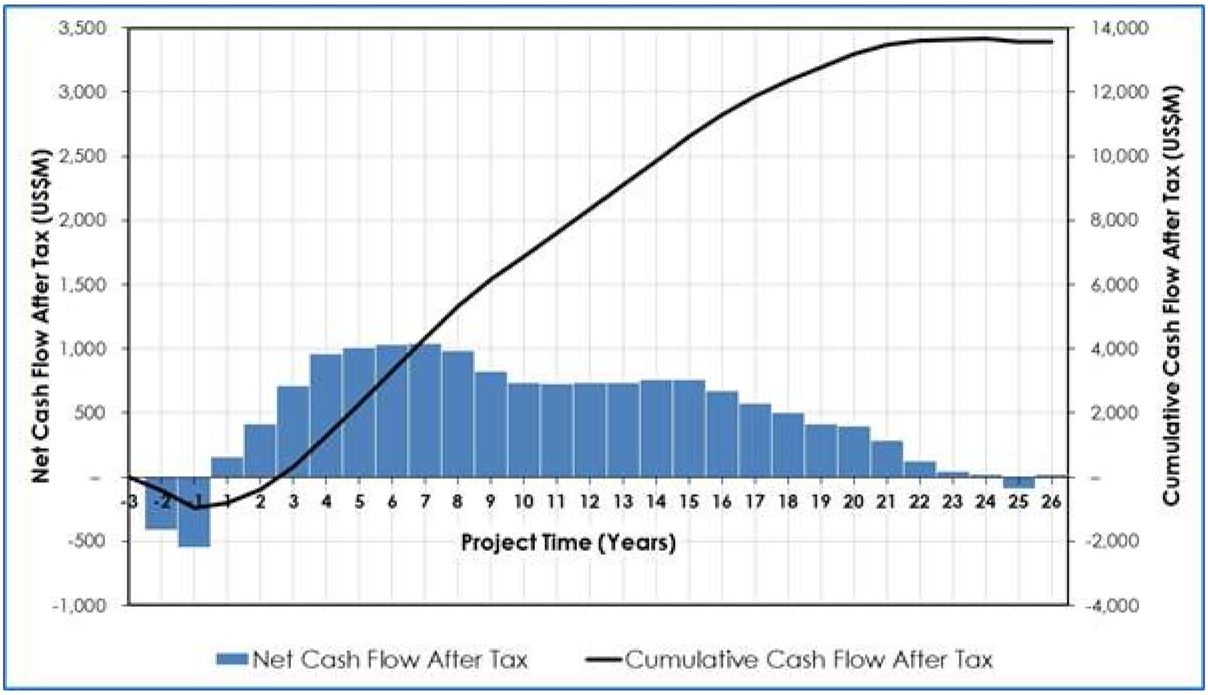


Figure by OreWin, 2019.

Table 22.9 Kakula Cash Flow 2019 PFS

Cash Flow Statement (US$M)	**Total**	**Year**									
Year Number		**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**11**	**21**
Year To									**10**	**20**	**LOM**
Gross Revenue	37,429	–	–	908	1,249	2,017	2,404	2,332	10,543	16,119	1,856
Realisation Costs	7,880	–	–	191	263	425	506	491	2,220	3,394	391
Net Revenue	**29,549**	**–**	**–**	**717**	**986**	**1,593**	**1,898**	**1,841**	**8,323**	**12,726**	**1,466**
Operating Costs											
Mining	4,585	–	–	67	127	167	214	213	1,084	2,225	487
Processing	1,549	–	–	36	45	68	73	73	362	711	181
Tailings	25	–	–	1	1	1	1	1	5	10	6
Smelter	–	–	–	–	–	–	–	–	–	–	–
General and Administration	816	–	–	16	32	32	32	35	174	336	160
Discount on Power	-212	-2	-4	-7	-9	-13	-14	-14	-75	-75	–
Customs Duties	347	–	–	6	10	13	16	16	82	167	37
Total Operating Costs	**7,111**	**-2**	**-4**	**120**	**207**	**268**	**322**	**324**	**1,632**	**3,374**	**871**
Operating Surplus / (Deficit)	**22,438**	**2**	**4**	**597**	**779**	**1,325**	**1,576**	**1,518**	**6,692**	**9,352**	**594**
Capital Costs											
Initial Capital	1,078	407	544	127	–	–	–	–	–	–	–
Expansion Capital	778	–	–	244	325	202	6	–	–	–	–
Sustaining Capital	1,295	–	–	–	10	10	161	115	329	495	176
Customs Duties	–	–	–	–	–	–	–	–	–	–	–
Working Capital	-31	12	14	69	28	62	33	-13	-60	-64	-112
Net Cash Flow Before Tax	**19,317**	**-417**	**-555**	**157**	**415**	**1,051**	**1,377**	**1,415**	**6,423**	**8,921**	**530**
Income Tax	5,741	–	–	–	–	338	418	409	1,803	2,635	137
Net Cash Flow After Tax	**13,575**	**-417**	**-555**	**157**	**415**	**712**	**959**	**1,006**	**4,619**	**6,285**	**393**

22.3 Kamoa 2019 PFS Overview and Results

The Kamoa 2019 PFS is an update of the work previously called the Kamoa 2017 PFS. The mine design, production, process plant and infrastructure have remained the same. Costs and price assumptions have been updated. Ivanhoe has developed twin declines at the Kansoko Mine on the Kansoko areas of the Kamoa deposit. Once in production, one will be a service decline for the transport of personnel and materials into the mine, and the second will be a conveyor decline for rock handling and transport of personnel and materials out of the mine. The Kansoko Mine has a Mineral Reserve that was previously stated in the Kamoa 2016 PFS and was updated in the Kamoa 2019 PFS.

The base case described in the Kamoa 2019 PFS is the construction and operation of an underground mine, concentrator processing facilities, and associated infrastructure. The Kamoa 2019 PFS production is planned to be an average of 6 Mtpa ore over a production period of 26 years.

The base case described in the Kamoa 2019 PFS is the construction and operation of a stand-alone underground mine, concentrator processing facilities, and associated infrastructure. The base case mining rate and concentrator feed capacity is 6 Mtpa. This refines the findings of the Kamoa March 2016 PFS, which envisaged a production rate of 3 Mtpa. The PFS is based entirely on the Kamoa 2019 PFS Mineral Reserve.

The Kamoa 2019 PFS re-assesses the development of the Kamoa Deposit as a stand-alone 6 Mtpa mining and processing complex. The LOM production scenario schedules 125.2 Mt to be mined at an average grade of 3.81% copper, producing 11.4 Mt of high-grade copper concentrate, containing approximately 9.2 billion pounds of copper.

The economic analysis uses a long-term price assumption of US$3.10/lb of copper and returns an after-tax NPV at an 8% discount rate of US$2.3 billion, an increase of 140% compared to the after-tax NPV 8% of US$986 million that was projected in the Kamoa 2016 PFS. It has an after-tax IRR of 26.7% and a payback period of 4.7 years. The LOM average mine site cash cost is US$0.66/lb of copper.

The estimated initial capital cost, including contingency, is US$1.0 billion. The capital expenditure for off-site power, which is included in the initial capital cost, includes a US$64M advance payment to the DRC state-owned electricity company, SNEL, to upgrade two hydropower plants (Koni and Mwadingusha) to provide the Kamoa-Kakula Project with access to clean electricity for its planned operations. Mwadingusha is being upgraded first. The work is being led by Stucky Ltd., of Switzerland; the advance payment will be recovered through a reduction in the power tariff once Kamoa is in operation. The key results of the Kamoa 2019 PFS are summarised in Table 22.10.

Table 22.10 Kamoa 2019 PFS Summary

Item	Unit	Total
Total Processed		
Quantity Milled	kt	125,182
Copper Feed Grade	%	3.81
Total Concentrate Produced		
Copper Concentrate Produced	kt (dry)	11,405
Copper Recovery	%	87.52
Copper Concentrate Grade	%	36.63
Contained Copper in Concentrate	Mlb	9,211
Contained Copper in Concentrate	kt	4,178
Peak Annual Contained Metal in Concentrate	kt	245
10 Year Average		
Copper Concentrate Produced	kt (dry)	487
Contained Copper in Concentrate	kt	178
Mine Site Cash Cost	US$/lb	0.59
Total Cash Cost	US$/lb	1.55
Key Financial Results		
Peak Funding	US$M	1,300
Initial Capital Cost	US$M	994
Expansion Capital Cost	US$M	299
Sustaining Capital Costs	US$M	1,359
LOM Average Mine Site Cash Cost	US$/lb Cu	0.66
LOM Average Total Cash Cost	US$/lb Cu	1.61
Site Operating Cost	US$/t Milled	46.71
After-Tax NPV8%	US$M	2,334
After-Tax IRR	%	26.7
Project Payback Period	Years	4.7
Initial Project Life	Years	26

Table 22.11 Kamoa 2019 PFS Financial Results

Net Present Value (US$M)	Discount Rate	Before Taxation	After Taxation
	Undiscounted	10,620	7,524
	4.0%	5,890	4,128
	6.0%	4,457	3,097
	8.0%	**3,399**	**2,334**
	10.0%	2,605	1,761
	12.0%	2,002	1,324
Internal Rate of Return		30.9%	26.7%
Project Payback Period (Years)		4.6	4.7

Table 22.12 Kamoa 2019 PFS Production and Processing

Item	Unit	Total LOM	Years 1–5	Years 1–10	LOM Average
Total Processed					
Quantity Milled	kt	125,182	3,533	4,777	4,815
Copper Feed Grade	%	3.81	4.24	4.20	3.81
Total Concentrate Produced					
Copper Concentrate Produced	kt (dry)	11,405	358	487	439
Copper Concentrate	kt (dry)	11,405	358	487	439
Copper Recovery	%	87.52	88.11	88.70	87.52
Copper Concentrate Grade	%	36.63	36.85	36.48	36.63
Contained Copper in Concentrate					
Copper	Mlb	9,211	291	392	354
Copper	kt	4,178	132	178	161
Payable Copper in Concentrate					
Copper	Mlb	8,884	280	378	342
Copper	kt	4,030	127	171	155
Payable Copper					
Copper	Mlb	8,884	280	378	342
Copper	kt	4,030	127	171	155

Figure 22.8 and Figure 22.9 depict the processing, concentrate and metal production, respectively. Table 22.13 summarises unit operating costs and Table 22.14 provides a breakdown of operating costs and revenue.

Figure 22.8 Kamoa 2019 PFS Process Production

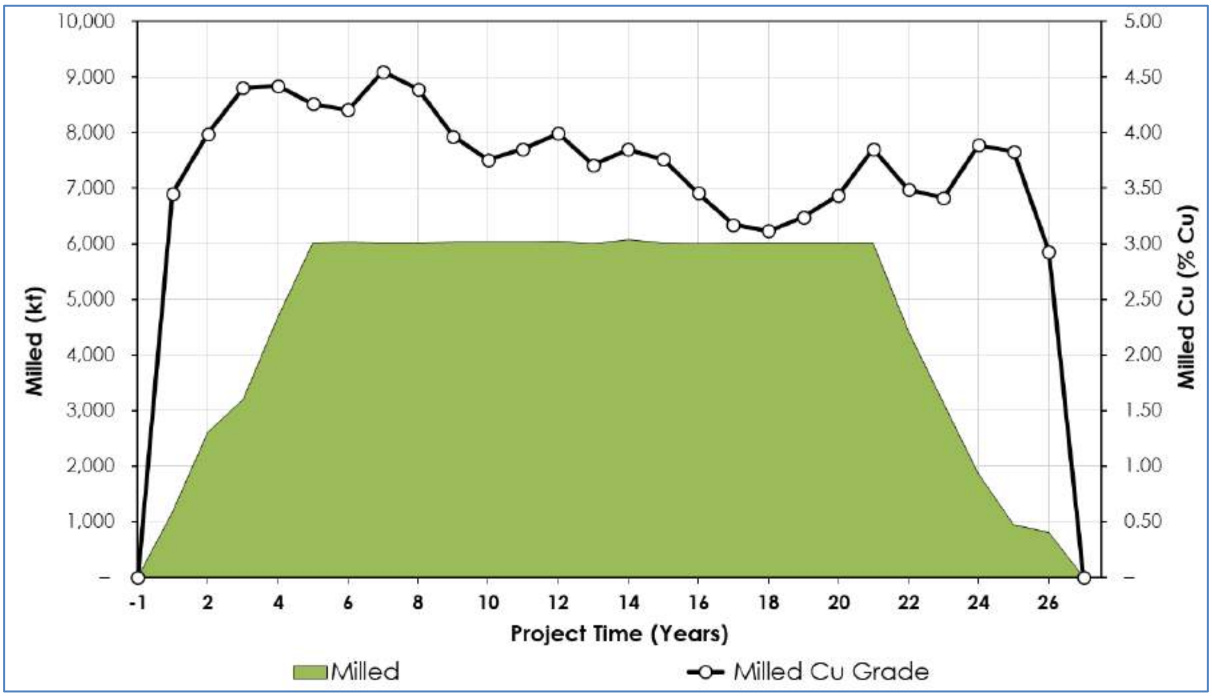


Figure by OreWin, 2019.

Figure 22.9 Kamoa 2019 PFS Concentrate and Metal Production

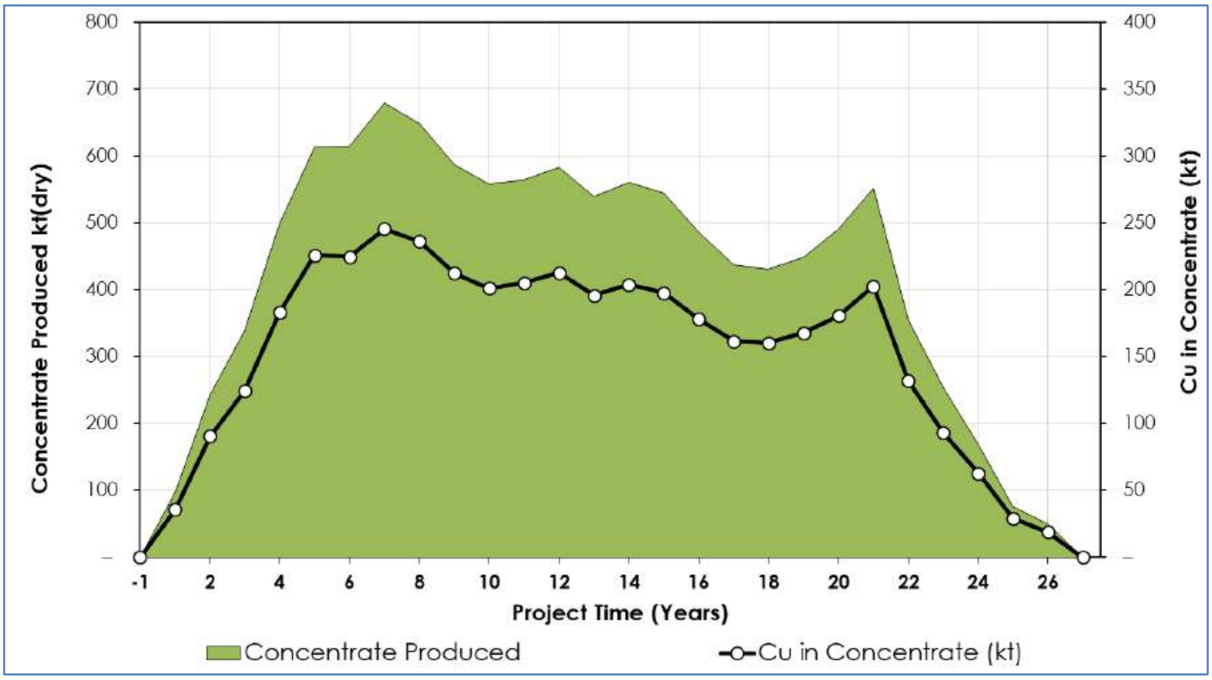


Figure by OreWin, 2019.

Table 22.13 Kamoa 2019 PFS Site Operating Costs

	Payable Cu (US$/lb)		
	Years 1–5	**Years 1–10**	**LOM Average**
Mine Site	0.64	0.59	0.66
Transport	0.51	0.51	0.51
Treatment and Refining Charges	0.18	0.19	0.19
Royalties and Export Tax	0.26	0.26	0.26
Total Cash Costs	**1.59**	**1.55**	**1.61**

Table 22.14 Kamoa 2019 PFS Revenue and Operating Costs

	Total LOM (US$M)	Years 1–5	Years 1–10	LOM Average
		US$/t Milled		
Revenue				
Copper in Concentrate	27,542	246.05	245.31	220.01
Gross Sales Revenue	**27,542**	**246.05**	**245.31**	**220.01**
Less: Realisation Costs				
Transport	4,521	40.14	40.45	36.12
Treatment and Refining	1,649	14.68	14.73	13.18
Royalties and Export Tax	2,287	20.36	20.42	18.27
Total Realisation Costs	**8,458**	**75.18**	**75.60**	**67.56**
Net Sales Revenue	**19,084**	**170.87**	**169.71**	**152.45**
Site Operating Costs				
Underground Mining	3,534	31.55	28.74	28.23
Processing	1,443	11.53	11.53	11.53
Tailings	29	0.31	0.24	0.24
Smelter	–	–	–	–
General and Administration	785	7.20	6.06	6.27
SNEL Discount	-231	-2.18	-2.12	-1.80
Customs Duties	281	2.46	2.29	2.25
Total	**5,842**	**50.86**	**46.74**	**46.71**
Net Operating Margin	**13,242**	**120.01**	**122.98**	**105.74**
Net Operating Margin	**69.39%**	**70.23%**	**72.46%**	**69.36%**

The capital costs for the project are detailed in Table 22.15.

Table 22.15 Kamoa 2019 PFS Capital Costs

Capital Costs (US$M)	Initial Capital US$M	Expansion Capital US$M	Sustaining Capital US$M	Total US$M
Mining				
Underground Mining	317	–	822	1,139
Capitalised Preproduction	13	–	–	13
Subtotal	**331**	**–**	**822**	**1,152**
Power and Smelter				
Power Supply Off Site	64	–	–	64
Subtotal	**64**	**–**	**–**	**64**
Concentrator and Tailings				
Process Plant	149	85	171	405
Tailings	21	97	–	118
Subtotal	**170**	**182**	**171**	**523**
Infrastructure				
Plant Infrastructure	113	–	85	197
Subtotal	**113**	**–**	**85**	**197**
Indirects				
EPCM	66	27	22	115
Owners Cost	80	27	9	116
Customs Duties	27	8	40	74
Closure	–	–	70	70
Subtotal	**173**	**62**	**140**	**376**
Capital Expenditure Before Contingency	**851**	**245**	**1,217**	**2,312**
Contingency	144	55	142	341
Capital Expenditure After Contingency	**994**	**299**	**1,359**	**2,653**

The cash flow sensitivity to metal price variation is shown in Table 22.16, for copper prices from US$2.00/lb Cu to US$4.00/lb.

The sensitivity of After Tax NPV$_8$ to initial capital cost, expansion capital cost, direct operating costs, treatment and refining, and transport are shown in Table 22.17. The table shows the change in the base case After Tax NPV$_8$ of US$2,334M. The sensitivity to treatment and refining applies the concentrate treatment charges of US$80/t concentrate and concentrate refining charge of US$0.08/lb Cu. The sensitivity to transport applies the costs via road (US$349/t) and via rail (US$254/t).

The change in Cu feed grade is approximately equivalent to a change in recovery or metal price because all three parameters are directly related to copper revenue.

Table 22.16 Kamoa Mine Copper Price Sensitivity

After Tax NPV (US$M)	**Copper Price - US$/lb**					
Discount Rate	**2.00**	**2.50**	**3.00**	**3.10**	**3.50**	**4.00**
Undiscounted	908	3,919	6,924	7,524	9,925	12,923
4.0%	155	1,979	3,771	4,128	5,556	7,338
6.0%	-74	1,389	2,813	3,097	4,230	5,643
8.0%	-241	953	2,105	**2,334**	3,249	4,388
10.0%	-363	626	1,573	1,761	2,510	3,443
12.0%	-452	379	1,168	1,324	1,947	2,721
15.0%	-542	113	726	848	1,331	1,930
IRR	5.3%	16.7%	25.1%	26.7%	32.4%	38.9%

Table 22.17 Kamoa Mine Additional Sensitivities

			Change from Base NPV$_8$% (US$M)				
Variable	**Units**	**Base Value**	**-25%**	**-10%**	**–**	**10%**	**25%**
Initial Capital Cost	US$M	994	2,508	2,403	2,334	2,264	2,160
Expansion Capital Cost	US$M	299	2,369	2,348	2,334	2,320	2,299
Initial and Expansion Capital Cost	US$M	1,294	2,543	2,418	2,334	2,250	2,124
Site Operating Cost	US$/t Milled	47	2,731	2,495	2,334	2,172	1,929
Treatment and Refining	US$/t and US$/lb Cu	80 / 0.08 / 110	2,443	2,378	2,334	2,290	2,224
Transport	US$/t Conc	349 / 254	2,637	2,455	2,334	2,213	2,031

The annual and cumulative cash flows are shown in Figure 22.10 (annual cash flow is shown on the left vertical axis and cumulative cash flow on the right axis). The Project cash flow is shown in Table 22.18.

Figure 22.10 Kamoa Projected Cumulative Cash Flow

Figure by OreWin, 2019.

Table 22.18 Kamoa Cash Flow 2019 PFS

Cash Flow Statement (US$M)	**Total**	**Year**									
Year Number		**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**11**	**21**
Year To									**10**	**20**	**LOM**
Gross Revenue	27,542	–	–	233	598	820	1,206	1,489	7,372	12,279	3,544
Realisation Costs	8,458	–	–	72	181	251	369	455	2,283	3,768	1,079
Net Revenue	**19,084**	**–**	**–**	**161**	**417**	**569**	**837**	**1,034**	**5,089**	**8,511**	**2,466**
Operating Costs											
Mining	3,534	–	–	28	72	115	148	195	816	1,672	490
Processing	1,443	–	–	13	30	37	54	69	347	694	198
Tailings	29	–	–	1	1	1	1	1	6	12	6
Smelter	–	–	–	–	–	–	–	–	–	–	–
General and Administration	785	–	–	14	21	30	30	33	162	319	176
Discount on Power	-231	-2	-4	-2	-5	-8	-11	-12	-63	-124	–
Customs Duties	281	–	–	2	6	9	11	15	66	134	38
Total Operating Costs	**5,842**	**-2**	**-4**	**56**	**125**	**184**	**232**	**301**	**1,334**	**2,707**	**908**
Operating Surplus / (Deficit)	**13,242**	**2**	**4**	**105**	**292**	**386**	**605**	**732**	**3,755**	**5,805**	**1,558**
Capital Costs											
Initial Capital	994	221	424	349	–	–	–	–	–	–	–
Expansion Capital	299	–	–	14	83	56	14	18	53	60	–
Sustaining Capital	1,359	–	–	–	210	192	98	71	270	346	173
Customs Duties	–	–	–	–	–	–	–	–	–	–	–
Working Capital	-31	-7	21	13	25	14	19	22	-19	-14	-105
Net Cash Flow Before Tax	**10,620**	**-212**	**-442**	**-271**	**-25**	**124**	**473**	**621**	**3,451**	**5,412**	**1,490**
Income Tax	3,096	–	–	–	–	–	–	124	958	1,596	417
Net Cash Flow After Tax	**7,524**	**-212**	**-442**	**-271**	**-25**	**124**	**473**	**496**	**2,493**	**3,816**	**1,073**

23 ADJACENT PROPERTIES

There are no adjacent properties relevant to this Report.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Kamoa-Kakula 2019 PEA

The Kamoa-Kakula 2019 PEA analyses a production case with an expansion of the Kakula concentrator processing facilities, and associated infrastructure to 18 Mtpa and includes a smelter and seven separate underground mining operations with associated capital and operating costs. The locations of the seven mines and the boundaries for the PFS and PEA cases are shown in Figure 24.1. The seven mines ranked by their relative values are:

- Kakula Mine (PFS 6 Mtpa).
- Kansoko Mine (PFS 6 Mtpa).
- Kakula West Mine (PEA 6 Mtpa).
- Kamoa Ouest Mine 1 (PEA 6 Mtpa).
- Kansoko Nord Mine 2 (PEA 6 Mtpa).
- Kamoa Centrale Mine 3 (PEA 6 Mtpa).
- Kamoa Nord Mine 4 (PEA 3 Mtpa).

Figure 24.1 Kamoa-Kakula IDP19 Mining Locations

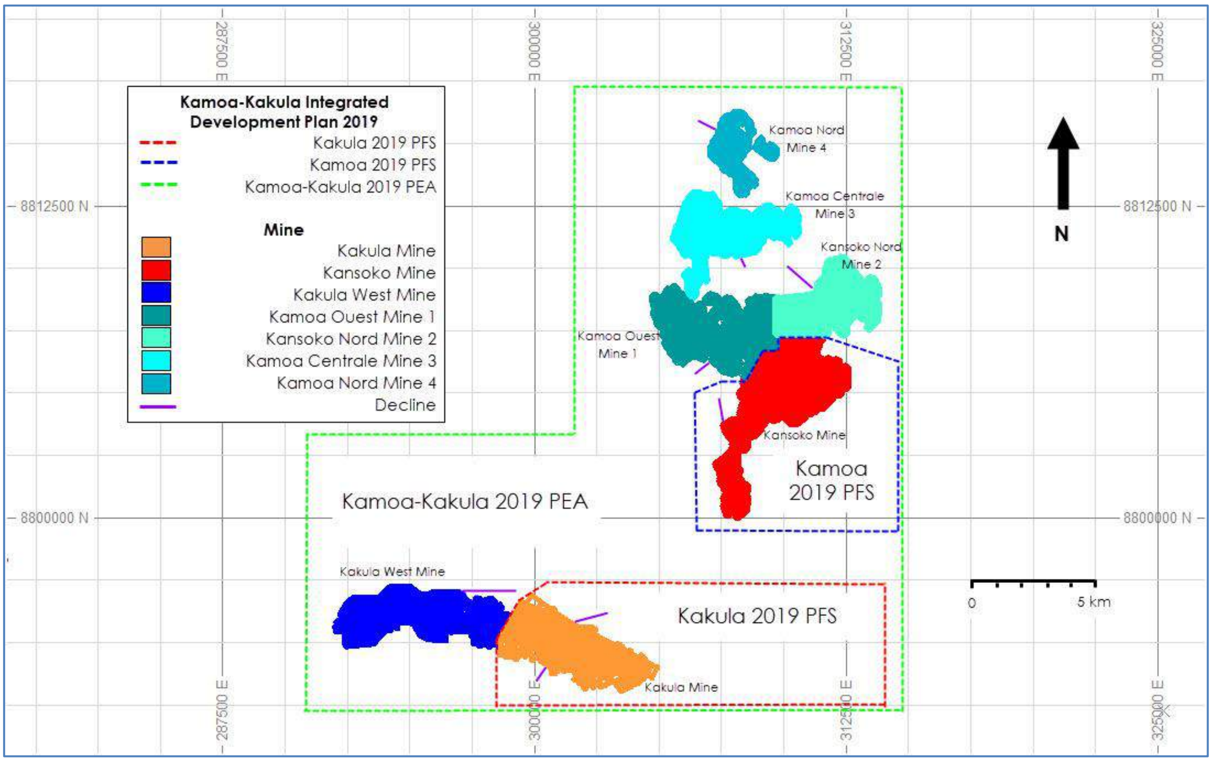


Figure by OreWin, 2019.

The Kamoa-Kakula 2019 PEA is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves – and there is no certainty that the results will be realised. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The Kamoa-Kakula 2019 PEA includes a PEA level study of the Kakula West. Kakula West is separated from Kakula by the West Scarp Fault and is planned as an independent mine. The Kakula West, Kamoa Ouest, Kansoko Nord, Kamoa Centrale and Kamoa Nord PEA analyses have been prepared using the Mineral Resources stated in the Kamoa-Kakula 2018 Resource Update. Since that time the Kakaula West Mineral Resource has been updated and the updated Mineral Resource has been stated in Section 14 of the Kamoa-Kakula IDP19.

The potential development scenarios at the Kamoa-Kakula Project include the Kamoa-Kakula IDP19 development scenario shown in Figure 24.2. The Kakula decline development is followed by the development of the stoping panels and construction of the plant. The initial plant capacity of 3 Mtpa is expanded to 6 Mtpa as the Kansoko Mine and Kakula Mine ramp up. The mines continue to ramp up to 12 Mtpa combined by Year 9. The next phase of development described by the Kamoa-Kakula 2019 PEA is from Kakula West to bring total production to 18 Mtpa this is then followed by this is then followed by four new mines at Kamoa North.

The immediate decisions for Ivanhoe and its partners are to determine the sequence for developing the initial operation.

Figure 24.2 Kamoa-Kakula IDP19 Long-Term Development Scenario

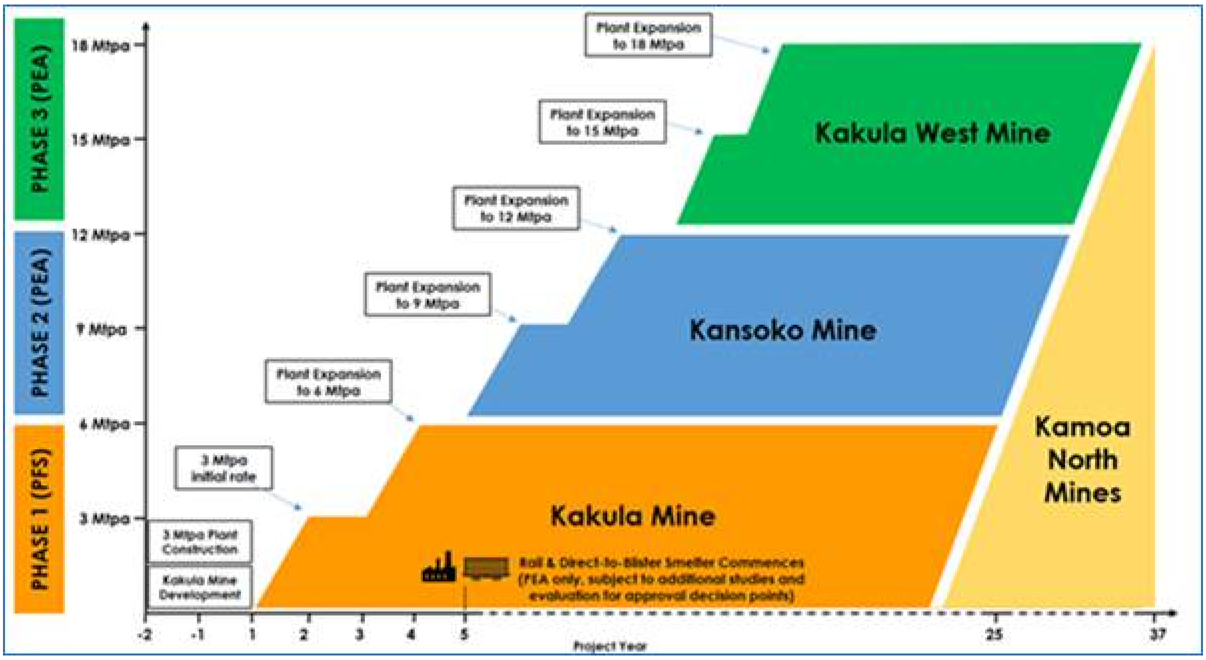


Figure by OreWin, 2019.

A site plan showing the locations of the mines and key infrastructure for Kakula and Kansoko mines is shown in Figure 24.3.

The Kamoa-Kakula 2019 PEA as part of the Kamoa-Kakula IDP19 includes economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability. The results of the Kamoa-Kakula 2019 PEA represent forward-looking information. The forward-looking information includes metal price assumptions, cash flow forecasts, projected capital and operating costs, metal recoveries, mine life and production rates, and other assumptions used in the Kamoa-Kakula 2019 PEA. Readers are cautioned that actual results may vary from those presented. The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially are presented in the body of this report under each relevant section.

Additional studies are required to evaluate feasibility and the timing of a higher plant feed from the Kakula mine, the Kansoko mine and the Kamoa North Mines of the Kamoa Deposit. Also, a sensitivity analysis is required to evaluate feasibility and the timing of an on-site smelter to produce blister copper at the mine site.

Figure 24.3 Kamoa-Kakula Integrated Development Plan 2019 Site Plan

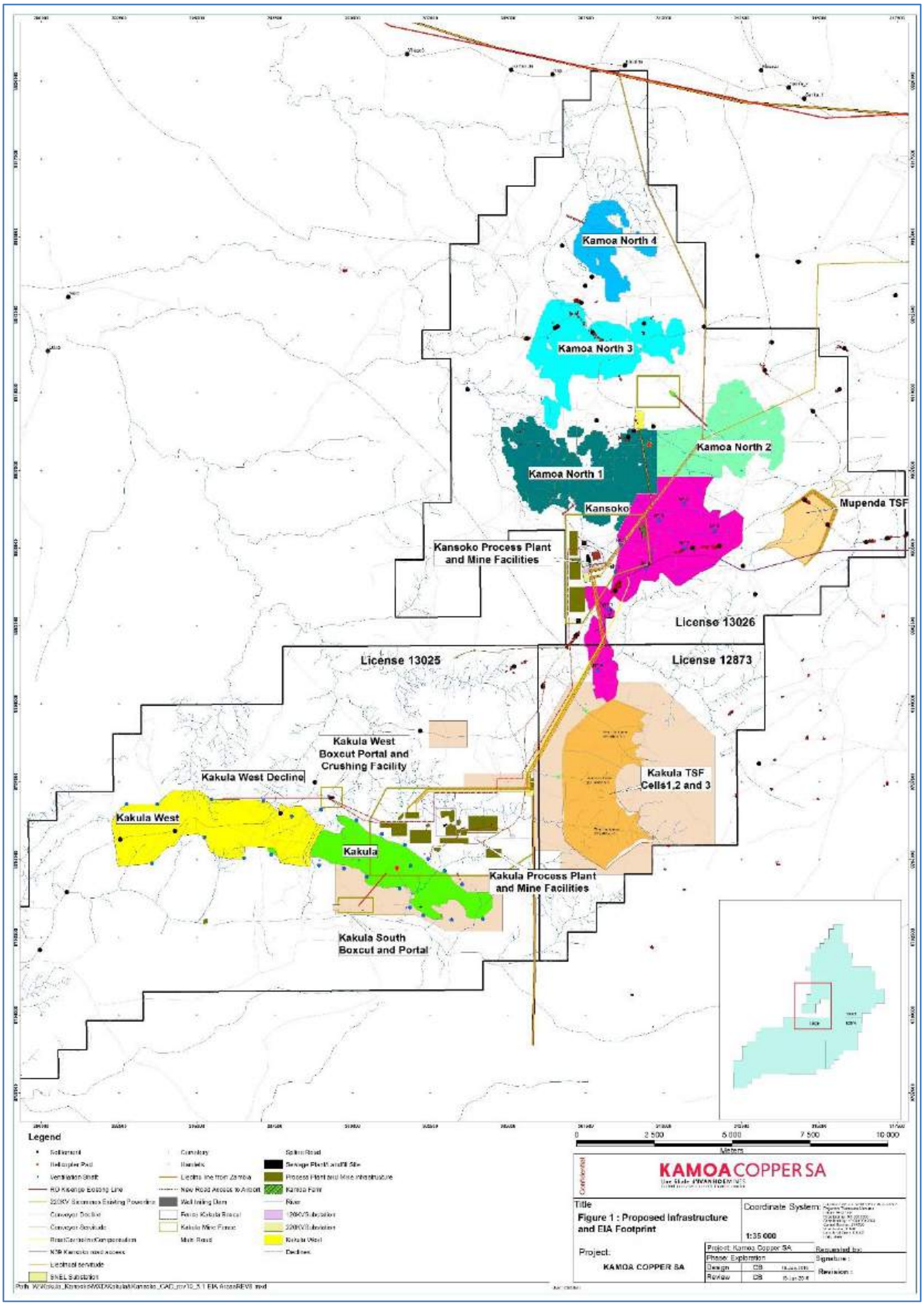


Figure by Kamoa Copper SA, 2019.

24.2 Kamoa-Kakula 2019 PEA Assumptions

24.2.1 Economic Assumptions

24.2.1.1 Pricing and Discount Rate Assumptions

The Project level valuation model begins on 31 December 2018. It is presented in 2018 constant dollars; cash flows are assumed to occur evenly during each year and a mid-year discounting approach is taken.

The copper price used for the evaluation is US$3.10/lb copper. This is considered to be reasonable based on industry forecasts and prices used in other studies. The product being sold is copper concentrate and payment terms for the copper assume that the LOM average payable copper concentrate is 97.73%.

The copper concentrate assumes an $80 per tonne treatment charge and refining charge of US$0.08/lb copper. The copper concentrate transport charge (including provincial road taxes and duties but excluding the provincial concentrate export tax and DRC export tax) to the customer is assumed to be US$349/t via road to Ndola and rail to Durban for shipping for the first two years of production and thereafter US$254/t via rail through Lobito.

24.2.1.2 Taxation

On 9 March 2018, Law No. 18/001 amending the 2002 Mining Code was promulgated. Kamoa Copper already adopted voluntarily several measures since the entry into force of the new legal framework governing mining activities in March 2018. For the purpose of this report, the economic analysis is based on the 2018 Mining Code. Kamoa Copper will continue to monitor the regulatory provisions to be adopted, ensuring as far as possible, continued adequate enforcement of the relevant legislative requirements. Detailed discussions are ongoing with the aim of resolving, in a fair and equitable manner, the mining industry’s concerns with the 2018 Mining Code.

The DRC Mining Code provides for all the taxes, charges, royalties, and other fees. Ivanhoe engaged KPMG South Africa, to report on which tax assumptions are applicable to an operating mine in the DRC. Only material taxes that would have an impact on the financial model have been considered and require confirmation.

In the analysis, carry balances such as tax and working capital calculations are based on nominal dollars and outputs are then deflated for use in the integrated cash flow calculation. The working capital assumptions for receivables, payables are 6 weeks and 6 weeks. These assumptions are preliminary and will need to be verified in later studies.

24.2.1.3 Royalties

A company holding a mining exploitation licence is subject to mining royalties. The royalty is due upon the sale of the product and is calculated at 3.5% 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 mining licence will 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.

24.2.1.4 Key Taxes

The DRC Mining Code provides for all the taxes, charges, royalties, and other fees. Ivanhoe engaged KPMG South Africa, to report on which tax assumptions are, applicable to an operating mine in the DRC. Only material taxes that would have an impact on the financial model have been considered and require confirmation. The key taxes identified by KPMG are listed below.

General Corporate Taxation

Companies that are the holders of mining rights are subject to tax at 30% on net income and withholding tax on distributions are subject to 10% tax at the shareholder's level. In addition, as from 01 January 2014, the minimum amount of tax payable by mining companies in a year is 1% of the calculated revenue for that specific year ("Minimum Tax Amount").

Funding / Thinning Capitalisation

No thin capitalisation rules apply in the DRC.

Tax Holidays

The DRC tax legislation does not currently provide for any tax holiday incentives.

Tax Losses

The aggregate exploration expenditure may be claimed.

Taxes on Products Sold

The tax rates will not change depending on whether concentrate or refined products are ultimately sold.

Depreciation

Specific mining assets dedicated to mining operations, with useful lives between 4 and 20 years are depreciated as follows:

- First year: 60% depreciated based on the cost of the asset.
- For subsequent years: a declining balance depreciation is applied based on the tax years remaining over the life of the mine.

Non-mining assets are depreciated in accordance with the common law. The common low provides different depreciation rates for various assets, e.g. 10 years for plant and equipment.

VAT

VAT came into effect in the DRC in January 2012. VAT is levied on all supplies of goods and services at a rate of 16% and is not levied on any capital asset movements.

Customs/Import Duties

Customs duty will be applied separately to capital (5%) and operating costs (10%) for direct cost line.

Export Taxes

National Export Tax

The fee is limited to 1% of the value of the export.

Provincial Export Tax on Concentrate

A provincial tax on the export of concentrate is levied on a per tonne basis and equates to US$100/t concentrate exported.

Provincial Export Road and Infrastructures Renovation Tax

A provincial export tax levied on any product exported by road is also levied on a per tonne basis at a rate of US$50/t. Copper concentrate will be exported by road to neighbouring countries and will thus be subject to the Road tax.

Withholding Taxes

A Withholding tax at the rate of 14% on services supplied by foreign companies established offshore to onshore companies applies. Mining companies are liable for movable property withholding tax at a rate of 10% in respect of dividends and other distributions paid. Non-mining companies are subject to withholding tax of 20%.

Dividend Distributions/Interest Repayments

Any dividend distributions made to Ivanhoe, as well as the DRC government will attract a withholding tax of 10%. A withholding tax of 20% applies if the loan is denominated in local DRC currency. If the loan is however denominated in foreign currency no withholding tax is payable. Interest payments to any local intermediate and holding companies attract a withholding tax of 20%.

Exceptional Tax on Expatriates

In the DRC, an employer is liable for the exceptional tax on expatriate's remuneration at a rate of 25%. Mining companies are subject to 10%. It is determined in terms of the salaries generated by the work carried out in the DRC and is deductible for purposes of calculating the income tax payable.

24.3 Kamoa-Kakula 2019 PEA Results

The Kamoa-Kakula 2019 PEA assesses an alternative development option of mining several deposits on the Kamoa-Kakula Project as an integrated, 18 Mtpa mining, processing and smelting complex, built in three stages. This scenario envisages the construction and operation of three separate mines: first, an initial 6 Mtpa mining operation would be established at the Kakula Mine on the Kakula Deposit; this is followed by a subsequent, separate 6 Mtpa mining operation at the Kansoko Mine using the existing twin declines that were completed in 2017; a third 6 Mtpa mine then will be established at the Kakula West Mine.

As the resources at the Kakula, Kansoko and Kakula West mines are mined out, production would begin sequentially at four other mines in the Kamoa North area to maintain throughput of 18 Mtpa to the then existing concentrator and smelter complex.

Each mining operation is expected to be a separate underground mine with a shared processing facility and surface infrastructure located at Kakula. Included in this scenario is the construction of a direct-to-blister flash copper smelter with a capacity of one Mt of copper concentrate per annum.

The development scenario of the Kakula Mine on the Kakula Deposit is shown in Figure 24.4.

Figure 24.4 Kamoa-Kakula IDP19 Long-Term Development Scenario

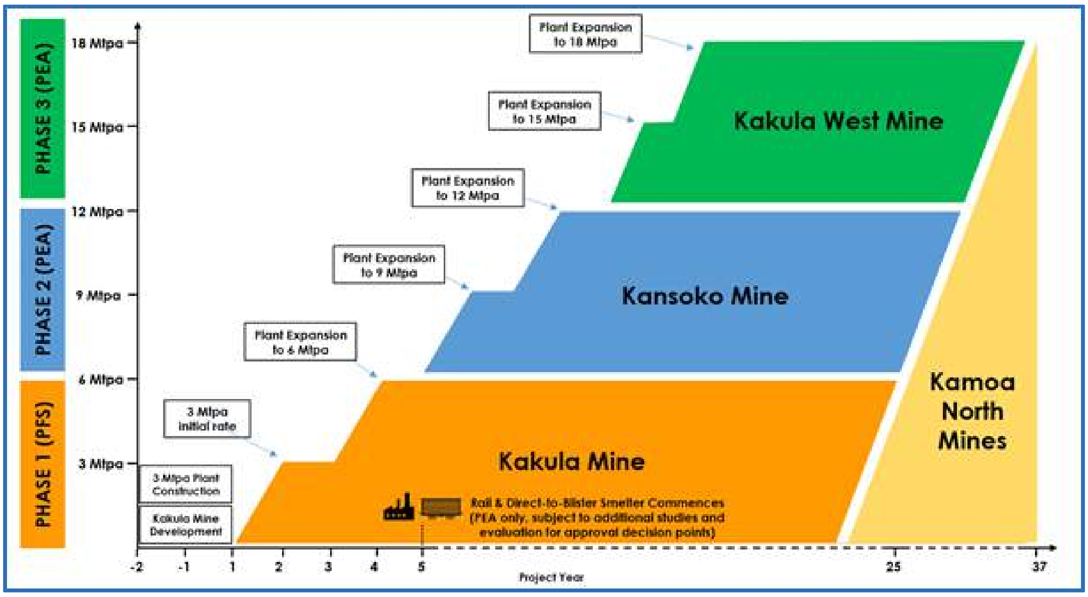


Figure by OreWin, 2019.

A summary of the key results for the Kamoa-Kakula 2019 PEA scenario are:

- Very-high-grade initial phase of production is projected to have a grade of 7.1% copper in second year of production and an average grade of 5.7% copper over the first 10 years of operations, resulting in estimated average annual copper production of 386,000 tonnes.
- Annual copper production is estimated at 360,000 tonnes in Year Four.
- Initial capital cost, including contingency, is estimated at US$1.1 billion.
- Average total cash cost of US$0.93/lb of copper during the first 10 years, including sulphuric acid credits.
- After-tax NPV, at an 8% discount rate, of US$10.0 billion.
- After-tax internal rate of return (IRR) of 40.95%, and a payback period of 2.9 years.

The LOM production scenario provides for 535.2 Mt to be mined at an average grade of 3.88% copper, producing 39 Mt of high-grade copper concentrate, containing approximately 38.5 billion pounds of copper.

The economic analysis uses a long-term price assumption of US$3.10/lb of copper and returns an after-tax NPV at an 8% discount rate of US$10.0 billion. It has an after-tax IRR of 40.9% and a payback period of 2.9 years.

The estimated initial capital cost, including contingency, is US$1.1 billion. The capital expenditure for off-site power, which is included in the initial capital cost, includes a US$64M advance payment to the DRC state-owned electricity company, SNEL, to upgrade two hydropower plants (Koni and Mwadingusha) to provide the Kamoa-Kakula Project with access to clean electricity for its planned operations. Mwadingusha is being upgraded first. The work is being led by Stucky Ltd., of Switzerland; the advance payment will be recovered through a reduction in the power tariff.

The Kamoa-Kakula 2019 PEA is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves—and there is no certainty that the results will be realised. Mineral Resources do not have demonstrated economic viability and are not Mineral Reserves.

Table 24.1 summarises the financial results. Key results of the Kamoa-Kakula 2019 PEA are summarised in Table 24.2. The mining production statistics are shown in Table 24.3. The Kamoa-Kakula 2019 PEA 18 Mtpa mill feed and copper grade profile for the LOM are shown in Figure 24.5 and the concentrate and metal production for the LOM are shown in Figure 24.6.

The Kamoa-Kakula 2019 PEA as part of the Kamoa-Kakula IDP19 includes economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability. The results of the Kamoa-Kakula IDP19 represent forward-looking information. The forward-looking information includes metal price assumptions, cash flow forecasts, projected capital and operating costs, metal recoveries, mine life and production rates, and other assumptions used in the Kamoa-Kakula IDP19. Readers are cautioned that actual results may vary from those presented. The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially are presented in the body of this report under each relevant section.

Table 24.1 Kamoa-Kakula 2019 PEA Financial Results

Net Present Value (US$M)	Discount Rate	Before Taxation	After Taxation
	Undiscounted	59,520	41,222
	4.0%	28,169	19,240
	6.0%	20,281	13,731
	8.0%	**14,967**	**10,030**
	10.0%	11,280	7,469
	12.0%	8,653	5,651
Internal Rate of Return	–	50.3%	40.9%
Project Payback Period (Years)	–	2.5	2.9

Table 24.2 Kamoa-Kakula 2019 PEA Results Summary for 18 Mtpa Production

Item	Unit	Total
Total Processed		
Quantity Milled	kt	535,217
Copper Feed Grade	%	3.88
Total Concentrate Produced		
Copper Concentrate Produced	kt (dry)	39,039
Copper Recovery	%	85.12
Copper Concentrate Grade	%	45.23
Contained Copper in Concentrate - External Smelter	Mlb	9,930
Contained Copper in Concentrate - External Smelter	kt	4,504
Contained Copper in Blister - Internal Smelter	Mlb	28,559
Contained Copper in Blister - Internal Smelter	kt	12,954
Peak Annual Recovered Copper Production	kt	740
10-Year Average		
Copper Concentrate Produced	kt (dry)	759
Contained Copper in Conc. - External Smelter	kt	121
Contained Copper in Blister - Internal Smelter	kt	261
Mine-Site Cash Cost	US$/lb Cu	0.63
Total Cash Cost	US$/lb Cu	0.93
Key Financial Results		
Peak Funding	US$M	1,099
Initial Capital Cost	US$M	1,078
Expansion Capital Cost	US$M	4,958
Sustaining Capital Cost	US$M	10,811
LOM Average Mine Site Cash Cost	US$/lb Cu	0.86
LOM Average Total Cash Cost	US$/lb Cu	1.10
Site Operating Cost	US$/t Milled	61.47
After-Tax NPV8%	US$M	10,030
After-Tax IRR	%	40.9
Project Payback Period	Years	2.9
Initial Project Life	Years	37

Table 24.3 Kamoa-Kakula 2019 PEA Production and Processing

Item	Unit	Total LOM	Years 1–5	Years 1–10	LOM Average
Total Processed					
Quantity Milled	kt	535,217	4,743	7,915	14,465
Copper Feed Grade	%	3.88	6.68	5.66	3.88
Total Concentrate Produced					
Copper Concentrate Produced	kt (dry)	39,039	478	759	1,055
Copper Concentrate - External Smelter	kt (dry)	8,491	344	211	229
Copper Concentrate - Internal Smelter	kt (dry)	30,549	135	549	826
Copper Recovery	%	85.12	85.62	86.18	85.12
Copper Concentrate Grade	%	45.23	56.76	50.82	45.23
Contained Copper in Concentrate - External Smelter					
Copper	Mlb	9,930	434	266	268
Copper	kt	4,504	197	121	122
Payable Copper in Concentrate - External Smelter					
Copper	Mlb	9,687	424	260	262
Copper	kt	4,394	192	118	119
Contained Copper in Blister - Internal Smelter					
Copper	Mlb	28,559	162	576	772
Copper	kt	12,954	73	261	350
Payable Copper in Blister - Internal Smelter					
Copper	Mlb	28,473	161	574	770
Copper	kt	12,915	73	260	349
Payable Copper					
Copper	Mlb	38,160	586	834	1,031
Copper	kt	17,309	266	378	468

Figure 24.5 Kamoa-Kakula 2019 PEA Process Production

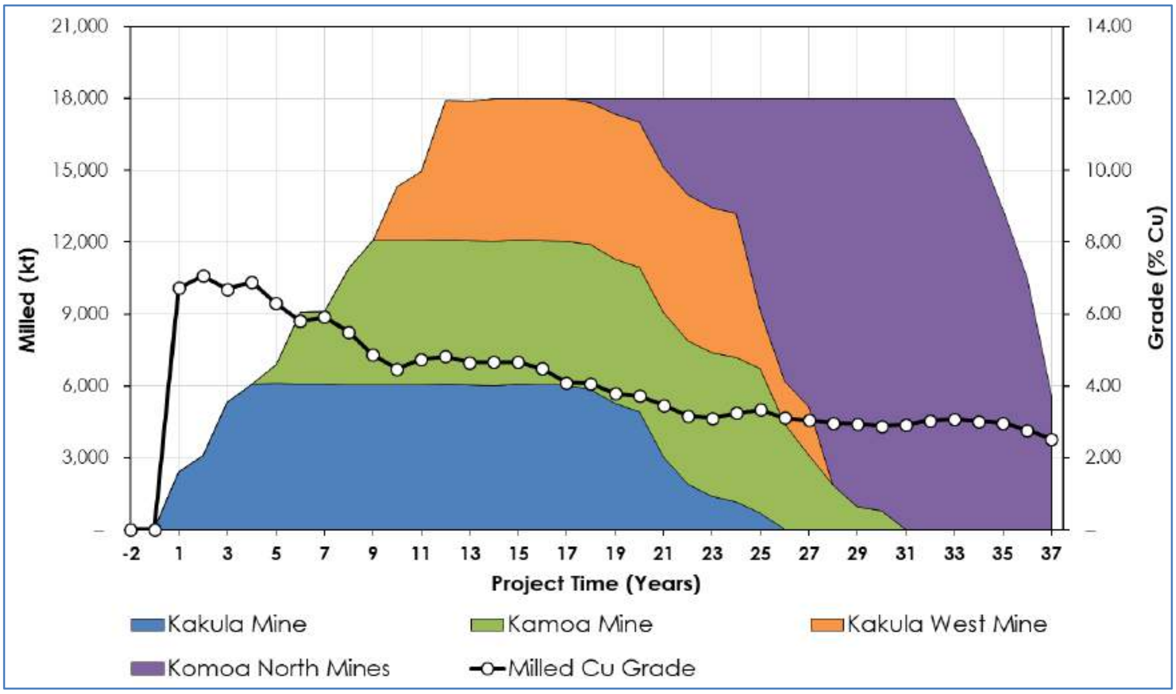


Figure by OreWin, 2019.

Figure 24.6 Kamoa-Kakula 2019 PEA Concentrate and Metal Production

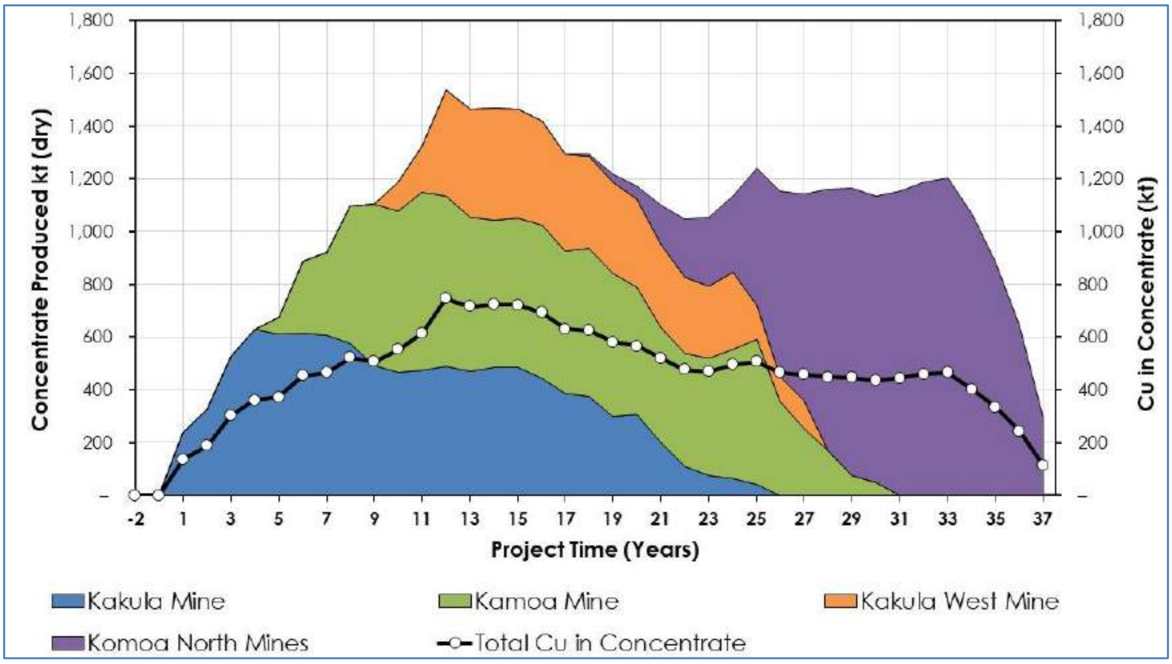


Figure by OreWin, 2019.

The Kamoa-Kakula 2019 PEA as part of the Kamoa-Kakula IDP19 includes economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability. The results of the Kamoa-Kakula IDP19 represent forward-looking information. The forward-looking information includes metal price assumptions, cash flow forecasts, projected capital and operating costs, metal recoveries, mine life and production rates, and other assumptions used in the Kamoa-Kakula IDP19. Readers are cautioned that actual results may vary from those presented. The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially are presented in the body of this report under each relevant section.

Table 24.4 summarises unit operating costs. Figure 24.7 compares the reported copper production in 2025 for the 20 highets producers by paid copper production. The Kamoa-Kakula 2019 PEA production is from the projected peak copper production which occurs in Year 12. Figure 24.8 shows the nominal paid copper production and head grade of the world 10 largest new greenfield projects. The estimates are based on public disclosure and information gathered by Wood Mackenzie. The Kamoa Kakula IDP19 was not reviewed by Wood Mackenzie prior to filing.

Table 24.4 Kamoa-Kakula 2019 PEA Unit Operating Costs

	Payable Copper (US$/lb)		
	Years 1–5	**Years 1–10**	**LOM Average**
Mine Site	0.46	0.53	0.75
Smelter	0.04	0.09	0.11
Transport	0.25	0.17	0.15
Treatment and Refining Charges	0.12	0.08	0.08
Royalties and Export Tax	0.18	0.16	0.16
Total Cash Costs	**1.05**	**1.04**	**1.25**
Sulphuric Acid Credits	0.03	0.11	0.15
Total Cash Costs After Credits	**1.02**	**0.93**	**1.10**

Figure 24.7 2025 Predicted World Copper Producer Production

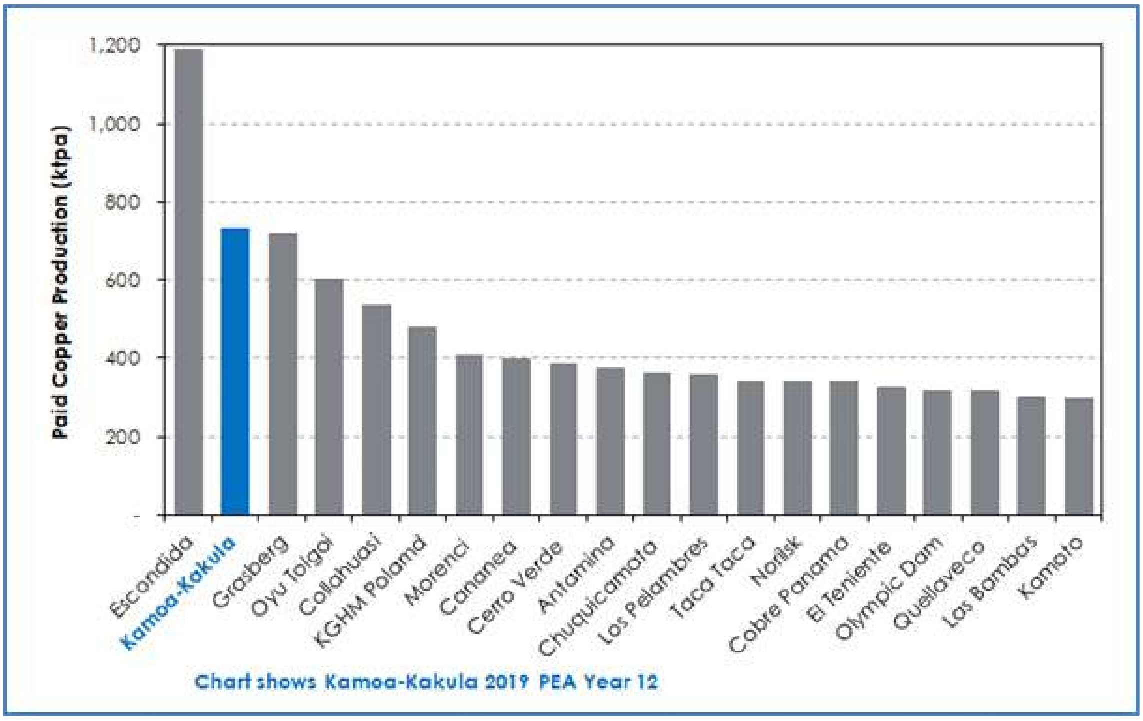


Figure by Ivanhoe, 2019. Source: Wood Mackenzie.

Figure 24.8 World Copper Producer Copper Productioon and Head Grade

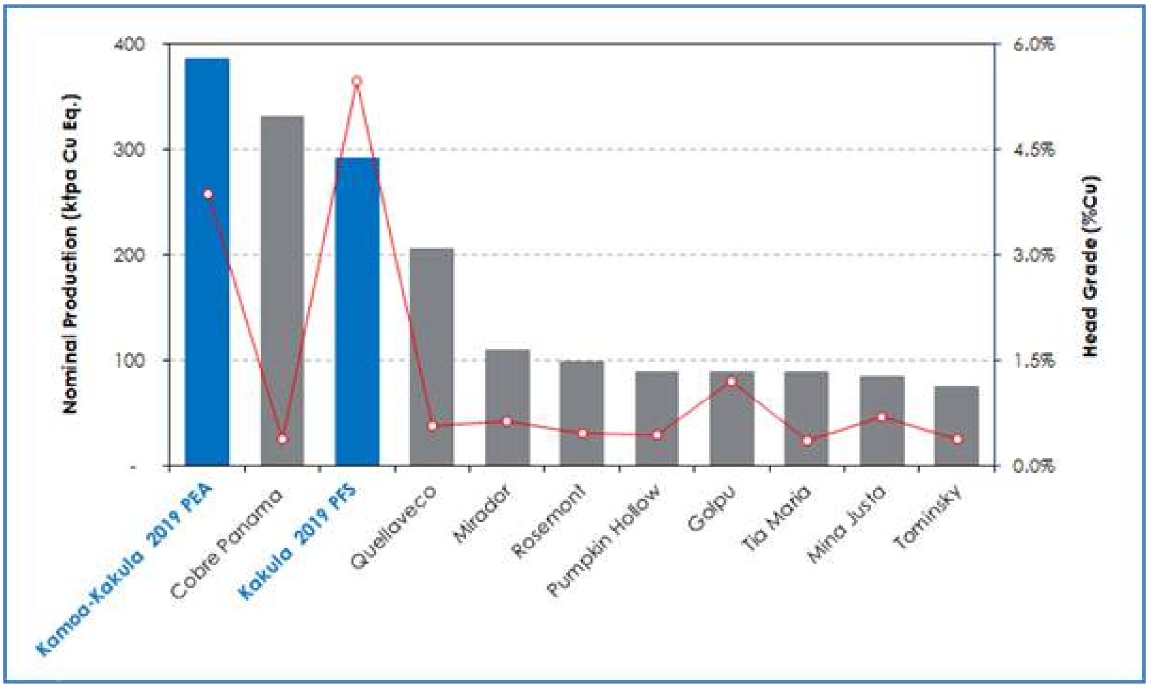


Figure by Ivanhoe, 2019. Source: Wood Mackenzie.

Table 24.5 provides a breakdown of revenue and operating costs. Capital costs for the project are detailed in Table 24.6.

Table 24.5 Kamoa-Kakula 2019 PEA Revenue and Operating Costs

	Total LOM US$M	Years 1–5	Years 1–10	LOM Average
		US$/t Milled		
Revenue				
Copper in Blister	88,266	105.43	224.77	164.92
Copper in Concentrate	30,029	277.36	101.89	56.11
Acid Production	5,786	3.46	11.25	10.81
Gross Sales Revenue	**124,081**	**386.25**	**337.90**	**231.83**
Less: Realisation Costs				
Transport	5,847	31.40	17.96	10.92
Treatment and Refining	2,899	14.82	8.45	5.42
Royalties and Export Tax	6,103	22.69	16.96	11.40
Total Realisation Costs	**14,848**	**68.90**	**43.36**	**27.74**
Net Sales Revenue	**109,233**	**317.34**	**294.54**	**204.09**
Site Operating Costs				
Underground Mining	18,513	35.12	35.29	34.59
Processing	7,350	15.27	14.97	13.73
Tailings	107	0.26	0.23	0.20
Smelter	4,117	4.52	9.99	7.69
General and Administration	1,750	6.25	5.32	3.27
SNEL Discount	-194	-3.11	-2.38	-0.35
Customs Duties	1,254	2.79	2.54	2.34
Total	**32,897**	**61.09**	**65.96**	**61.47**
Net Operating Margin	**76,337**	**256.25**	**228.58**	**142.62**
Net Operating Margin	**69.88%**	**80.75%**	**77.60%**	**69.88%**

Table 24.6 Kamoa-Kakula 2019 PEA Capital Costs

Description	Initial Capital US$M	Expansion Capital US$M	Sustaining Capital US$M	Total US$M
Mining				
Underground Mining	287	753	6,216	7,256
Capitalised Preproduction	107	–	–	107
Subtotal	**394**	**753**	**6,216**	**7,364**
Power and Smelter				
Smelter Total	–	770	716	1,487
Power Supply Off Site	64	–	–	64
Capitalised Power Cost	–	–	–	–
Subtotal	**64**	**770**	**716**	**1,551**
Concentrate and Tailings				
Process Plant	190	863	950	2,003
Tailings	24	53	167	244
Subtotal	**214**	**916**	**1,116**	**2,246**
Infrastructure				
Plant Infrastructure	109	756	707	1,573
General Infrastructure	–	353	204	557
Contractor's and Owner's Camps	–	–	73	73
Rail Link	–	72	–	72
Subtotal	**109**	**1,181**	**985**	**2,275**
Indirects				
EPCM	56	147	93	297
Owners Cost	103	323	–	426
Customs Duties	29	141	326	496
Closure	–	–	362	362
Subtotal	**188**	**611**	**782**	**1,581**
Capital Expenditure Before Contingency	**968**	**4,233**	**9,816**	**15,017**
Contingency	110	725	995	1,831
Capital Expenditure After Contingency	**1,078**	**4,958**	**10,811**	**16,847**

The after-tax NPV sensitivity to metal price variation is shown in Table 24.7 for copper prices from US$2.00/lb to US$4.00/lb.

The annual production results are shown in Table 24.9 to Table 24.11. The annual and cumulative cash flows are shown in Figure 24.9 (annual cash flow is shown on the left vertical axis and cumulative cash flow on the right axis).

Table 24.7 Kamoa-Kakula 2019 PEA Copper Price Sensitivity

After-Tax NPV (US$M)	**Copper Price - US$/lb**					
Discount Rate	**2.00**	**2.50**	**3.00**	**3.10**	**3.50**	**4.00**
Undiscounted	13,117	25,902	38,668	41,222	51,435	64,154
4.0%	5,684	11,850	18,008	19,240	24,165	30,307
6.0%	3,788	8,311	12,828	13,731	17,341	21,845
8.0%	2,510	5,931	9,347	**10,030**	12,758	16,164
10.0%	1,627	4,286	6,939	7,469	9,587	12,231
12.0%	1,005	3,120	5,229	5,651	7,332	9,433
15.0%	385	1,942	3,493	3,803	5,037	6,579
IRR	18.0%	28.9%	39.0%	**40.9%**	48.3%	57.0%

Figure 24.9 Kamoa-Kakula 2019 PEA Projected Cumulative Cash Flow

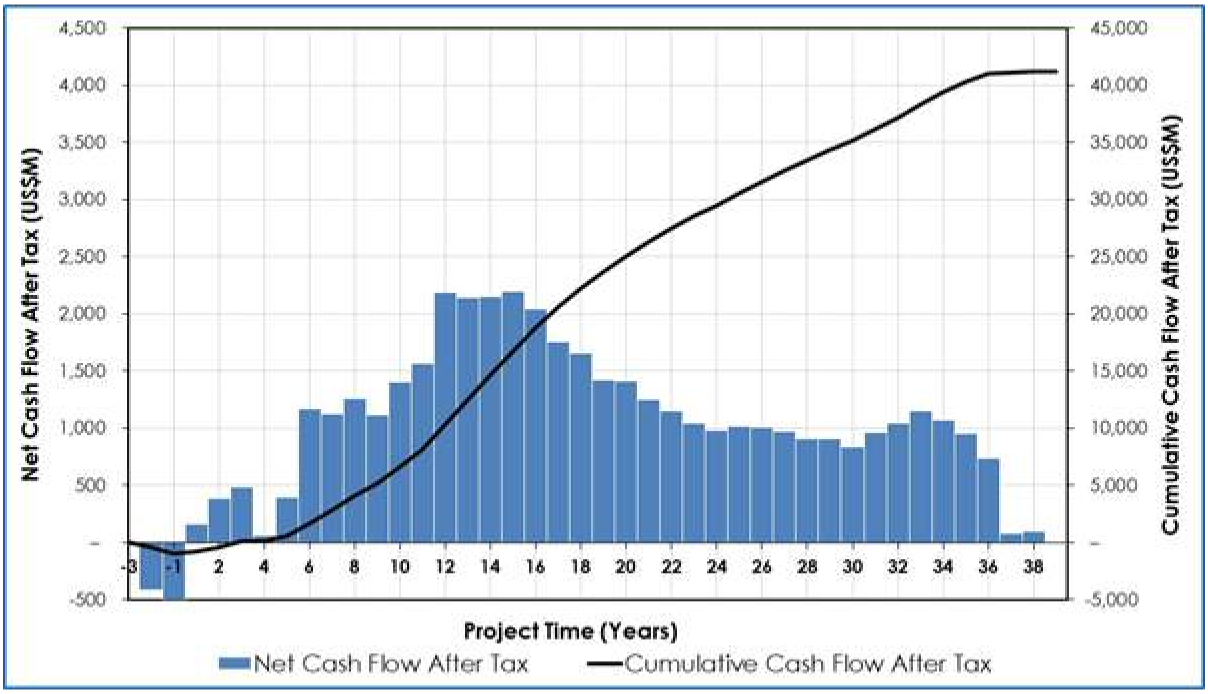


Figure by OreWin, 2019.

Table 24.8 Cash Flow – Kamoa-Kakula 2019 PEA

Cash Flow Statement (US$M)	Total	Year									
Year Number		**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**11**	**21**
Year To									**10**	**20**	**LOM**
Gross Revenue	124,081	–	–	908	1,249	2,017	2,404	2,582	17,585	46,161	51,176
Realisation Costs	14,848	–	–	191	263	425	506	249	1,798	5,601	5,815
Net Revenue	**109,233**	**–**	**–**	**717**	**986**	**1,593**	**1,898**	**2,333**	**15,787**	**40,560**	**45,361**
Operating Costs											
Mining	18,513	–	–	67	127	167	219	252	1,960	6,323	9,396
Processing	7,350	–	–	38	50	67	98	109	823	2,395	3,770
Tailings	107	–	–	1	1	1	1	2	12	36	52
Smelter	4,117	–	–	–	–	–	–	107	683	1,365	1,962
General & Administration	1,750	–	–	16	32	32	32	37	273	631	699
Discount on Power	-194	-2	-4	-7	-8	-12	-15	-32	-114	–	–
Customs Duties	1,254	–	–	6	10	13	18	18	135	421	632
Total Operating Costs	**32,897**	**-2**	**-4**	**121**	**212**	**267**	**354**	**494**	**3,772**	**11,170**	**16,511**
Operating Surplus / (Deficit)	**76,337**	**2**	**4**	**596**	**773**	**1,325**	**1,544**	**1,839**	**12,015**	**29,389**	**28,850**
Capital Costs											
Initial Capital	1,078	407	544	127	–	–	–	–	–	–	–
Expansion Capital	4,958	–	–	110	196	253	856	859	1,261	422	1,001
Sustaining Capital	10,811	–	–	128	163	176	176	138	1,845	2,869	5,316
Customs Duties	–	–	–	–	–	–	–	–	–	–	–
Working Capital	-31	12	14	68	32	82	99	42	72	-23	-430
Net Cash Flow Before Tax	**59,520**	**-417**	**-555**	**162**	**384**	**814**	**412**	**800**	**8,836**	**26,122**	**22,962**
Income Tax	18,298	–	–	–	–	329	354	409	2,779	7,595	6,831
Net Cash Flow After Tax	**41,222**	**-417**	**-555**	**162**	**384**	**485**	**58**	**391**	**6,057**	**18,526**	**16,131**

Table 24.9 Processing Production Schedule – Kamoa-Kakula 2019 PEA

Description	Units	Totals	Project Time (Years)														
			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Quantity Milled	kt	**535,217**	2,400	3,075	5,302	6,067	6,871	9,064	9,088	10,920	12,050	14,312	14,951	17,927	17,874	18,000	18,000
Cu Feed Grade	% Cu	**3.88**	6.73	7.06	6.70	6.90	6.30	5.81	5.91	5.51	4.87	4.47	4.74	4.82	4.65	4.66	4.66
Fe Feed Grade	% Fe	**5.48**	4.61	4.79	4.67	4.70	4.85	5.17	5.22	5.49	5.90	5.73	5.55	5.35	5.27	5.29	5.31
As Feed Grade	% As	**0.00**	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
S Feed Grade	% S	**1.83**	1.76	1.85	1.73	1.77	1.75	1.95	2.09	2.14	2.21	2.14	2.17	1.91	1.70	1.79	1.86
Copper Conc. Produced	kt (dry)	**39,039**	237	326	527	628	673	889	924	1,096	1,105	1,187	1,325	1,538	1,466	1,469	1,466
Copper Conc. - External Smelter	kt (dry)	**8,491**	237	326	527	628	–	–	–	96	105	187	325	538	466	469	466
Copper Conc. -Internal Smelter	kt (dry)	**30,549**	–	–	–	–	673	889	924	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000
Copper Concentrate Recovery	%	**85.12**	84.21	86.06	85.06	86.00	86.02	86.21	86.45	86.80	86.60	86.37	86.85	86.41	86.15	86.16	86.04
Copper Concentrate Grade	% Cu	**45.23**	57.32	57.32	57.32	57.32	55.32	51.09	50.28	47.59	45.96	46.56	46.46	48.52	48.85	49.24	49.25
Contained Copper in Conc. - External Smelter	Mlb	**9,930**	300	412	666	793	–	–	–	122	132	236	411	680	589	592	589
Contained Copper in Conc. - External Smelter	kt	**4,504**	136	187	302	360	–	–	–	55	60	107	186	308	267	269	267
Contained Copper in Blister - Internal Smelter	Mlb	**28,559**	–	–	–	–	809	986	1,009	1,013	972	967	932	951	975	987	988
Contained Copper in Blister - Internal Smelter	kt	**12,954**	–	–	–	–	367	447	457	460	441	439	423	431	442	448	448
Total Recovered Copper Production	Mlb	**38,489**	300	412	666	793	809	986	1,009	1,135	1,105	1,204	1,343	1,631	1,564	1,579	1,577
Total Recovered Copper Production	kt	**17,458**	136	187	302	360	367	447	457	515	501	546	609	740	710	716	715

Table 24.10 Processing Production Schedule – Kamoa-Kakula 2019 PEA

Description	Units	Project Time (Years)															
		16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31
Quantity Milled	kt	18,000	18,000	18,000	18,000	18,000	18,000	18,000	18,000	18,000	18,000	18,000	18,000	18,000	18,000	18,000	18,000
Cu Feed Grade	% Cu	4.48	4.09	4.06	3.79	3.72	3.46	3.17	3.10	3.24	3.34	3.11	3.05	2.97	2.94	2.87	2.93
Fe Feed Grade	% Fe	5.32	5.36	5.37	5.33	5.44	5.42	5.39	5.46	5.32	5.28	5.28	5.31	5.36	5.46	5.63	5.86
As Feed Grade	% As	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
S Feed Grade	% S	1.82	1.68	1.67	1.61	1.64	1.68	1.63	1.70	1.65	1.79	1.76	1.61	1.48	1.56	1.68	1.89
Copper Conc. Produced	kt (dry)	1,421	1,296	1,296	1,219	1,174	1,101	1,046	1,055	1,133	1,240	1,154	1,143	1,161	1,166	1,135	1,155
Copper Conc. - External Smelter	kt (dry)	421	296	296	219	174	101	46	55	133	240	154	143	161	166	135	155
Copper Conc. -Internal Smelter	kt (dry)	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000	1,000
Copper Concentrate Recovery	%	85.88	85.64	85.35	85.10	84.42	83.45	83.56	84.11	84.92	84.30	83.14	83.52	83.67	84.20	84.14	84.49
Copper Concentrate Grade	% Cu	48.77	48.61	48.20	47.58	48.16	47.22	45.59	44.51	43.77	40.83	40.30	40.09	38.52	38.25	38.33	38.51
Contained Copper in Conc. - External Smelter	Mlb	532	374	374	277	220	128	58	69	168	279	176	167	146	145	116	129
Contained Copper in Conc. - External Smelter	kt	241	169	169	126	100	58	26	31	76	126	80	76	66	66	53	59
Contained Copper in Blister - Internal Smelter	Mlb	981	1,000	988	987	1,011	1,003	978	951	912	824	837	830	827	826	830	839
Contained Copper in Blister - Internal Smelter	kt	445	453	448	448	459	455	444	432	413	374	380	376	375	375	376	380
Total Recovered Copper Production	Mlb	1,513	1,373	1,362	1,264	1,231	1,131	1,037	1,021	1,079	1,103	1,013	997	973	971	946	968
Total Recovered Copper Production	kt	686	623	618	573	559	513	470	463	490	500	459	452	441	440	429	439

Table 24.11 Processing Production Schedule – Kamoa-Kakula 2019 PEA

Description	Units	Project Time (Years)															
		32	33	34	35	36	37	38	39	40	41	42	43	44	45	46	47
Quantity Milled	kt	18,000	18,000	15,914	13,361	10,488	5,552	–	–	–	–	–	–	–	–	–	–
Cu Feed Grade	% Cu	3.04	3.07	3.02	2.96	2.76	2.51	–	–	–	–	–	–	–	–	–	–
Fe Feed Grade	% Fe	6.06	5.94	5.92	6.07	6.23	6.51	–	–	–	–	–	–	–	–	–	–
As Feed Grade	% As	0.00	0.00	0.00	0.00	0.00	0.00	–	–	–	–	–	–	–	–	–	–
S Feed Grade	% S	2.18	2.11	1.95	2.17	2.35	2.54	–	–	–	–	–	–	–	–	–	–
Copper Conc. Produced	kt (dry)	1,187	1,202	1,067	888	648	296	–	–	–	–	–	–	–	–	–	–
Copper Conc. - External Smelter	kt (dry)	187	202	200	177	164	296	–	–	–	–	–	–	–	–	–	–
Copper Conc. -Internal Smelter	kt (dry)	1,000	1,000	868	711	484	–	–	–	–	–	–	–	–	–	–	–
Copper Concentrate Recovery	%	84.02	83.90	84.06	84.49	83.96	80.02	–	–	–	–	–	–	–	–	–	–
Copper Concentrate Grade	% Cu	38.68	38.61	37.79	37.64	37.56	37.76	–	–	–	–	–	–	–	–	–	–
Contained Copper in Conc. - External Smelter	Mlb	164	176	173	151	139	246	–	–	–	–	–	–	–	–	–	–
Contained Copper in Conc. - External Smelter	kt	74	80	79	68	63	112	–	–	–	–	–	–	–	–	–	–
Contained Copper in Blister - Internal Smelter	Mlb	836	835	705	578	392	–	–	–	–	–	–	–	–	–	–	–
Contained Copper in Blister - Internal Smelter	kt	379	379	320	262	178	–	–	–	–	–	–	–	–	–	–	–
Total Recovered Copper Production	Mlb	1,000	1,011	878	728	531	246	–	–	–	–	–	–	–	–	–	–
Total Recovered Copper Production	kt	453	459	398	330	241	112	–	–	–	–	–	–	–	–	–	–

24.4 Kamoa-Kakula 2019 PEA Mining

The Kamoa-Kakula 2019 PEA analyses a production case with an expansion of the Kakula concentrator processing facilities, and associated infrastructure to 18 Mtpa and includes a smelter and seven separate underground mining operations with associated capital and operating costs. The locations of the seven mines and the boundaries for the PFS and PEA cases are shown in Figure 24.1. The seven mines ranked by their relative values are:

- Kakula Mine (PFS 6 Mtpa).
- Kansoko Mine (PFS 6 Mtpa).
- Kakula West Mine (PEA 6 Mtpa).
- Kamoa Ouest Mine 1 (PEA 6 Mtpa).
- Kansoko Nord Mine 2 (PEA 6 Mtpa).
- Kamoa Centrale Mine 3 (PEA 6 Mtpa).
- Kamoa Nord Mine 4 (PEA 3 Mtpa).

Mining methods in the Kamoa-Kakula 2019 PEA are assumed to be a combination of the controlled convergence room-and-pillar mining method, drift-and-fill with paste fill mining method, and room-and-pillar mining method.

At Kakula Mine the main mining method is drift-and-fill as described in the Kakula 2019 PFS, at the Kansoko Mine the main mining method is controlled convergence room-and-pillar method. At Kakula West there is a combination of drift-and-fill and controlled convergence room-and-pillar. Selection of the mining method was dictated by mining height and dip. The controlled convergence room-and-pillar method was selected for heights greater than 3 m and less than 6 m, and dip less than 25°. The drift-and-fill with paste fill was selected for heights greater than 6 m. The drift-and-fill with paste fill method was also selected for heights greater than 3 m and less than 6 m, and dip greater than 25°. At the Kamoa North Mines the mining method selected is controlled convergence room-and-pillar.

Figure 24.10 shows the locations of the seven mines and the boundaries for the PFS and PEA cases in IDP19.

Figure 24.10 Kamoa-Kakula 2019 PEA Development and Mining Zones

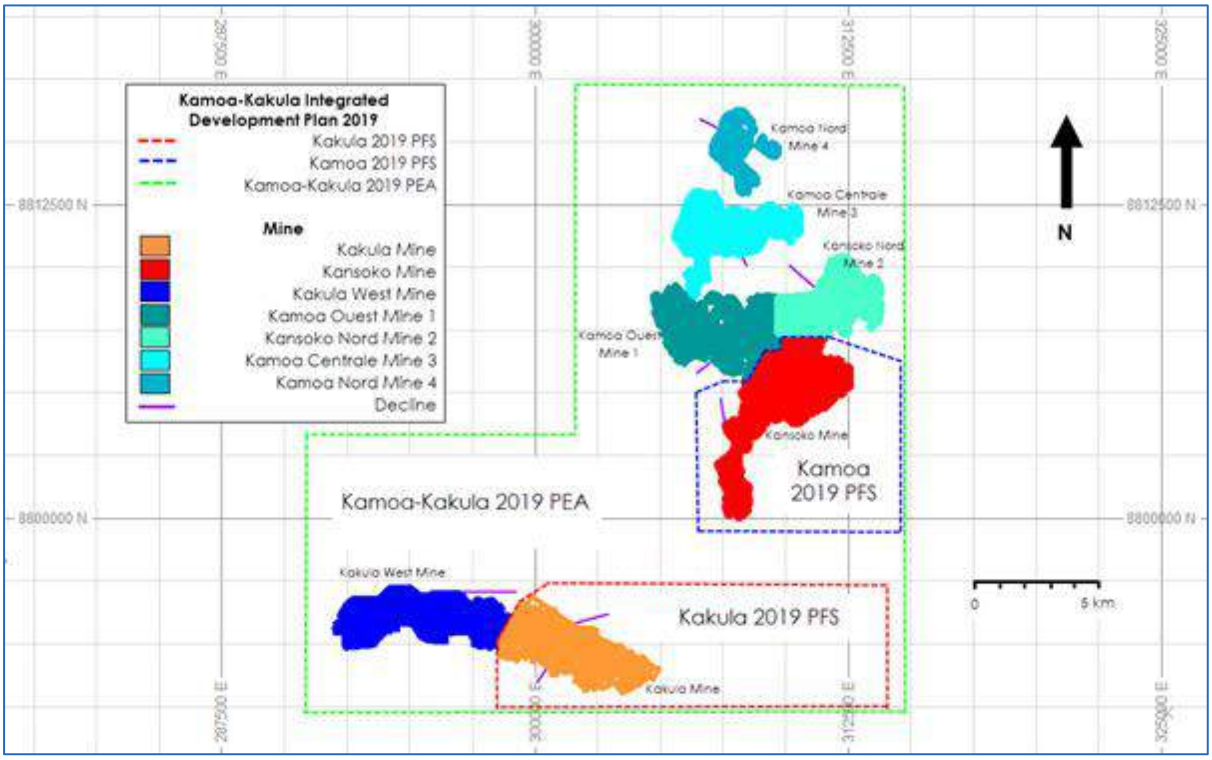


Figure by OreWin, 2019.

24.4.1 Kakula West Geotechnical Investigation and Design

This section contains a summary of the PEA-level mining geotechnical investigation and design completed by Ivanhoe mines (new horizons) for the Kakula West PEA. This summary follows below:

The geotechnical investigation was based on geotechnical drilling and logging completed by Ivanhoe mines over the Kakula West project area. A basic preliminary assessment was undertaken during this assessment and no core testing of the rock was completed. It was decided that a reasonable indication of the rock geotechnical properties obtained from the Kakula 2019 PFS area and this could be extrapolated into the Kakula West project area for PEA purposes. However, point load strength test were done during the core logging procedure and these are discussed it this summary.

Geotechnical Database

Geotechnical drilling and logging specific to the Kakula West area were conducted by Ivanhoe Mines.

24.4.1.1 Geotechnical and structural model

Structural Model

An assessment of the structural model completed by Ivanhoe that based on structural logging data obtained from acoustic televiewer (ATV) measurement for 4 holes in the Kakula West area. The following observations were made:

- Two joint sets were identified as dominant with a north-west and a south-east orientation in the hangingwall. A similar set of joints were identified in the footwall.
- The orebody was found to have a zone of sub-horizontal bedding dipping to the north-west. A secondary joint set was also identified similar to the with a south-east orientation.

24.4.1.2 Geotechnical Assessment

For the geotechnical assessment Kamoa identified three consistent lithological intervals for each borehole and these are:

- Sandstone in the footwall, with approximately 10 m thickness;
- Diamictite / Siltstone in the mineralized zone, with approximately 5–10 m thickness;
- Diamictite / Siltstone in the hangingwall, with approximately 30–40 m thickness.

Rock Strength Properties

No laboratory test work was completed for this PEA, however the results obtained from the Kakula 2019 PFS were extrapolated and used here. Point Load Testing was however done on the geotechnical core and the results for this exercise are shown in Table 24.12.

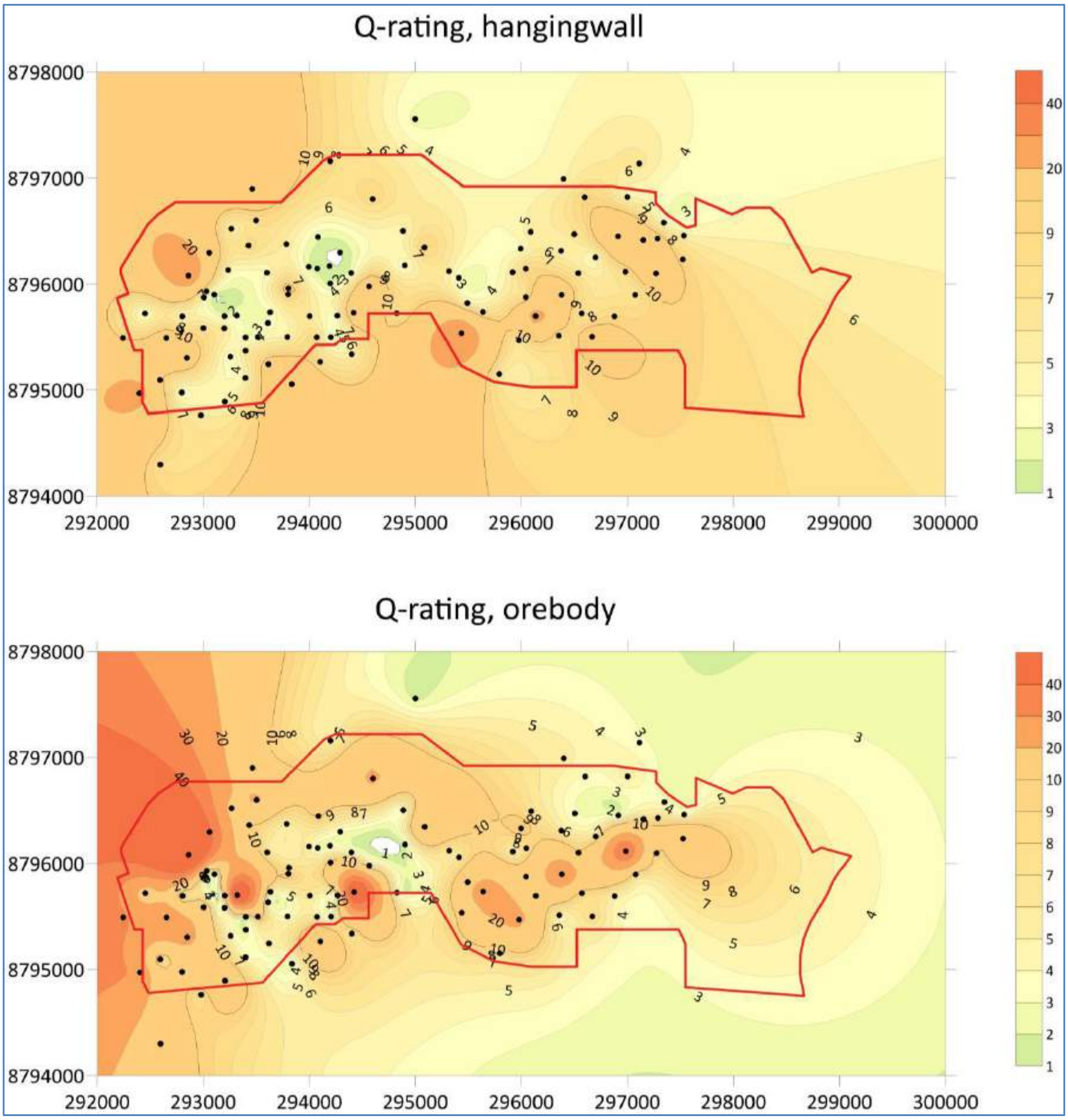
Table 24.12 UCS Results Obtained from Point Load Tests

Lithology Units	Count of Samples	Min of ICS (MPa)	Average of UCS (MPa)	Max of UCS (MPa)	Std Dev of UCS
SDT – diamictite	343	32.6	72.9	125.7	23
SSL – Siltstone	140	49.5	88.3	138.2	24
SST - sandstone	179	46.6	101.3	202.2	39

Rock Mass Properties

The rock mass properties at Kakula were inferred from geotechnical logging data from 100 drillholes. The logging of the drillholes was carried out by mine personnel at the site. The Laubscher's 1990 rock mass classification system was used for this logging exercise. Ivanhoe then completed a direct conversion of the parameters obtained to obtain the Barton's Q system. These results are shown in Figure 24.11.

Figure 24.11 Q-Rating Calculated from the Drill Core Geotechnical Logging Data



24.4.1.3 Geotechnical Risk Associated With The Kakula West

It was found that the orebody consisted of fair to good quality rock mass in the central and eastern areas. For the hangingwall more poor quality rock mass conditions were identified specifically in the western side of the project. Highly fractured zones were identified associated with steeply dipping zones in the western area.

It is expected that more complicated ground stability problems will be encountered on the western side.

No laboratory test work has been completed and this should be rectified during the PFS stage of this study in the Kakula West property.

A second rock mass classification system should be implemented in the PFS to confirm the results obtained in this study.

No water assessment has been completed for this area.

24.4.1.4 Preliminary Mining Method Recomendations

The following recommendations are made for Kakula:

- Extraction ratios may need to be reduced.
- Drift-and-fill mining will be required for most areas.
- A single cut option with a reduced excavation spans is recommended.
- Mining sequence should be optimised in addition to a reduced advance rate.

Support systems that cater for the expected ground conditions should be implemented.

24.4.2 Kakula West Mining

The West Scarp Fault was used to define the Kakula West resource model from the Kakula resource model. Figure 24.23 shows the Kakula West resource model area and the West Scarp Fault in the middle of the Kakula resource model. The preliminary mineable area was obtained using a stope shape optimiser (applied to the Kakula West resource model. Stope optimisation was undertaken on the resource model at mining cut-off grades 2.50% Cu. A dilution allowance of 30 cm on footwall and hangingwall was added to the model. The resulting stope shapes were then further optimised to select a suitable mining method for the area with a height more than 6 m.

The proposed mining methods for Kakula West were assumed to be a combination of controlled convergence room-and-pillar mining method and drift-and-fill with paste fill mining method based on the Kakula 2019 PEA study.

Two mining methods were chosen:

- Controlled convergence room-and-pillar.
- Drift-and-fill with paste fill.

The criteria to select the mining methods for the stope were the height and the dip of the optimised stope. The controlled convergence room-and-pillar method was selected for heights greater than 3 m and less than 6 m and dip less than 25°. The drift-and-fill with paste fill was selected for heights greater than 6 m. The drift-and-fill with paste fill method was also selected for heights greater than 3 m and less than 6 m and dip greater than 25°.

The controlled convergence room-and-pillar method (3 m to 6 m high) allows in-panel pillars to be stripped so the backs and floors can converge in a controlled manner meaning no backfill is required. The protection pillars between the mine workings and preparatory workings are successively extracted as the mining front progresses. Typical extraction ratios are shown in Table 24.13.

Figure 24.12 Kakula West Resource Model Area

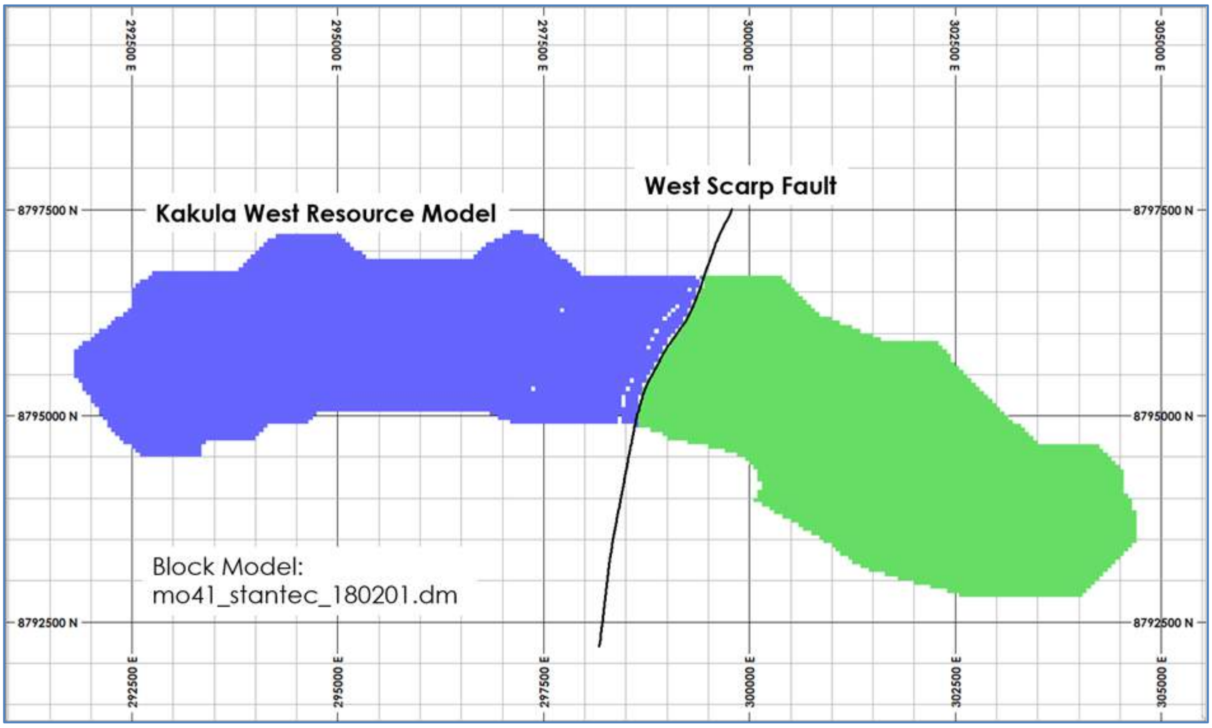


Figure by OreWin, 2019.

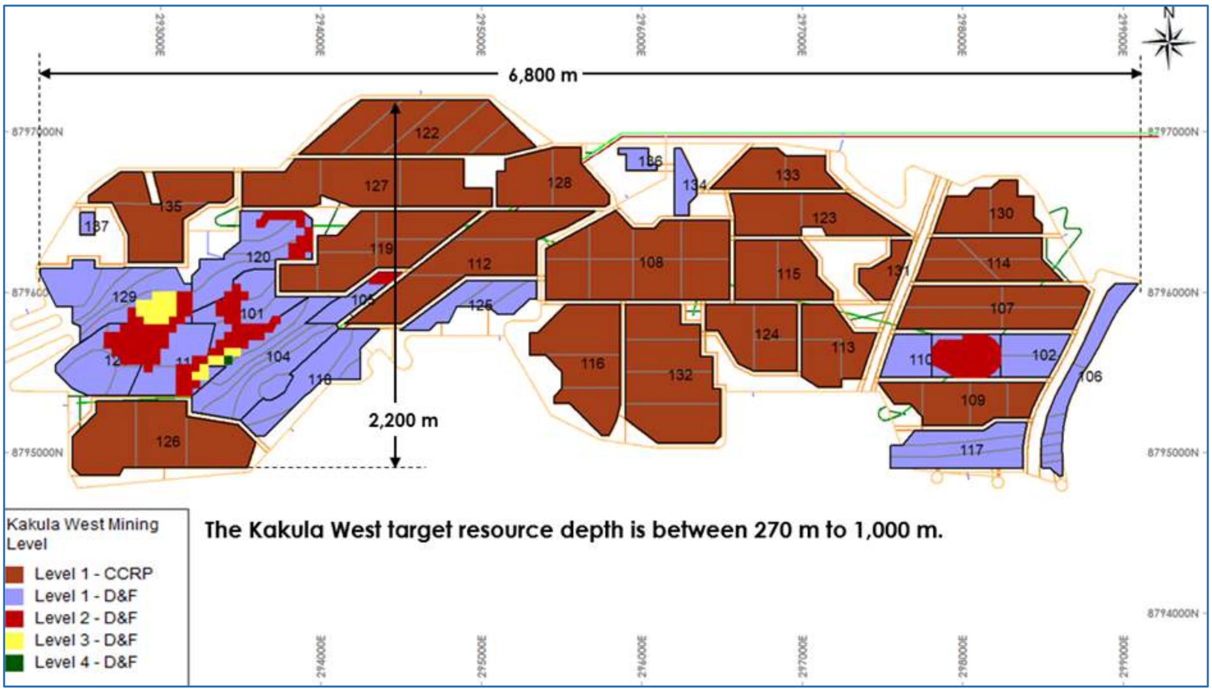
Table 24.13 Controlled Convergence Room-and-Pillar in Panel Extraction

Deposit Dip	Thickness (mining height)	Extraction Ratio for Mining Panel
up to 12°	3–6 m	90.00%
up to 16°	3-6 m	85.00%
up to 25°	3-6 m	80.00%

Drift-and-fill mining method is a selective underground mining method and ideal for steeply dipping high-grade deposits. The drift-and-fill mining panels would be mined in a primary, secondary, and tertiary sequence and an extraction of 85% was calculated for the drift-and-fill panels.

The Kakula West area dimensions is 6.80 km by 2.20 km and the depth of the Kakula West targeted resource is between 270 m to 1,000 m. Access will be via the twin declines to the mining zones on the north side of the Kakula West deposit. Selection of the mining method was dictated by mining height and dip and the size of the mining zones. The Kakula West mining zones, mining method and the twin decline are shown in Figure 24.13. It is assumed that the controlled convergence room-and-pillar mining zones will be protected by a 20 m pillar. The main access to the controlled convergence room-and-pillar zones is from the main development service access as shown in Figure 24.13. The drift-and-fill mining zones will be filled and protected by paste fill. The main access to the drift-and-fill zones is from the developments underneath the drift-and-fill zones and the main development service access. The extraction ratio of the drift-and-fill zones for all lifts are 85% and the mined tonnes and grades are diluted by 3.00% paste fill.

Figure 24.13 Kakula West 2019 PEA Mining Zones and Twin Declines



The decline was designed to accommodate two parallel drives dipping at -18%, one for personnel and machinery, the other for a conveyor. Material mined will be hauled to the transfer points and then transported by conveyor to surface.

The conveyor decline measures 7.0 m (W) x 6.0 m (H) and the service decline measures 5.5 m (W) x 6.0 m (H). The conveyor and service declines are spaced 13.25 m apart. Every 80 m down decline, a 13.25 m cross-cut between the declines and twin remuck cubbies are required. The conveyor drive dips at -18% run E-W with dimensions 7.0 m (W) x 6.0 m (H). Personnel and machinery access measures 5.5 m (W) x 6.0 m (H) and does not exceed a dip of 18%.

Mine ventilation is achieved through eight upcast 5.0 m ventilation raises and four 5.0 m ventilation shafts in varying intake and exhaust combinations depending on the location of mining and air movement requirement. The first ventilation raise will be developed through the declines, then a series of the ventilation shafts and raise will be developed to provide the fresh air for the production panels and development area. The ventilation shafts will decrease the mine production ramp up period and increase the production to feed the plant at 6 Mtpa as soon as possible.

Backfill boreholes were strategically designed and located to supply backfill to the drift-and-fill panels over the life of the project.

The Kakula West 2019 PEA development design and mining zones are shown in Figure 24.14. The ventilation raise and shaft locations are shown in Figure 24.15.

Figure 24.14 Kakula West 2019 PEA Development

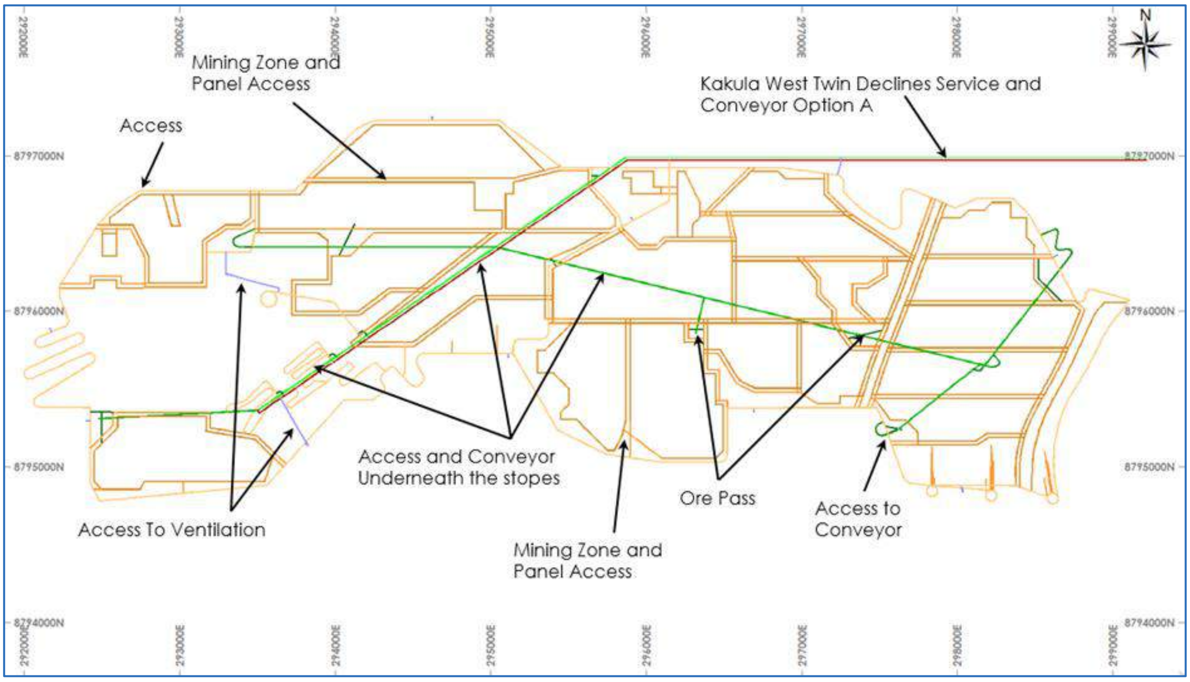
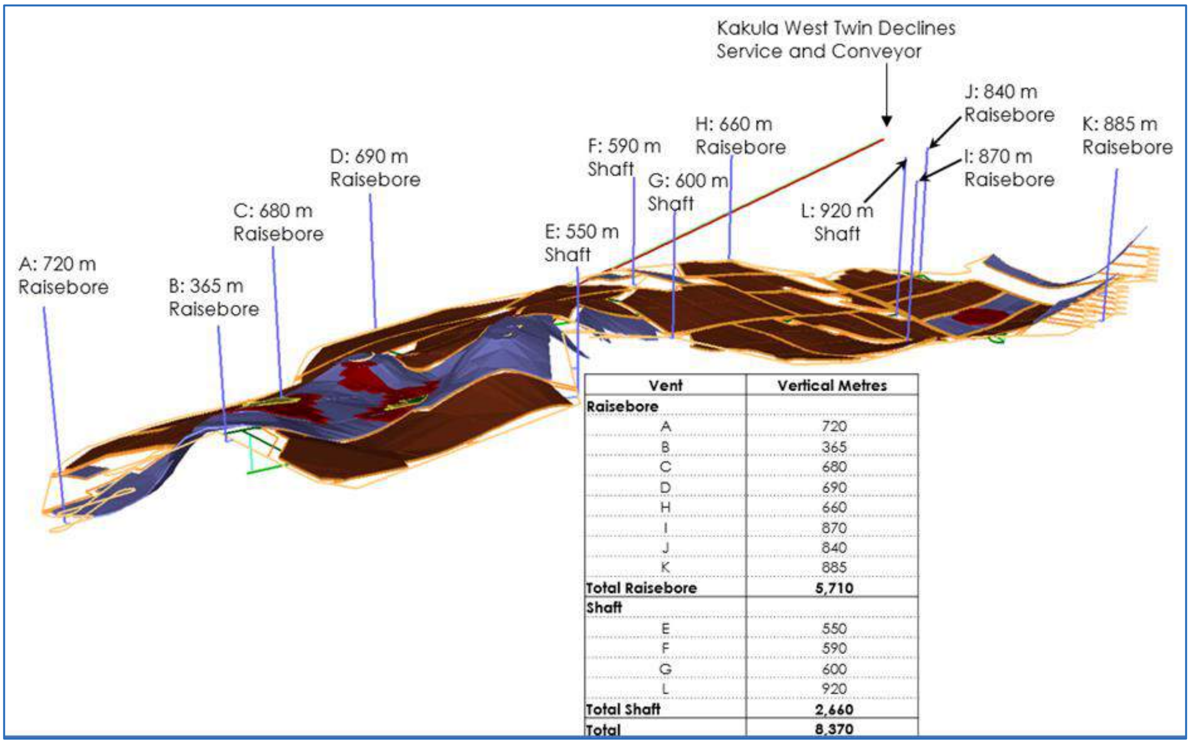


Figure by OreWin, 2019.

Figure 24.15 Kakula West 2019 PEA Ventilation Location



Vent	Vertical Metres
Raisebore	
A	720
B	365
C	680
D	690
H	660
I	870
J	840
K	885
Total Raisebore	**5,710**
Shaft	
E	550
F	590
G	600
L	920
Total Shaft	**2,660**
Total	**8,370**

Figure by OreWin, 2019.

A Kakula West Mineral Resource of approximately 89.0 Mt at 3.99 % Cu has been defined on multiple mining zones. The mine production rate is 6 Mtpa and the life of mine is 19 years. The 6 Mtpa production rate is based on the preliminary Kakula West mine development and production case study. Mining costs were developed using the contractor mining costs from the current development at the Kansoko Mine, and factored fixed costs and unit rates from the Kamoa 2019 PFS.

24.5 Kamoa North Mines

The Kamoa North deposit is located North of the Kansoko mine and consists of four separate mines. The Kamoa North mines are based on a preliminary UG optimisation at $ $80/t NSR14 cut-off grade. The Resources model name is kam14a160309.dm. The study assesses the development and production of the Kamoa North deposit at a maximum of 18 Mtpa underground mine production from four separate mines. The locations of the four Kamoa North mines and the boundaries for the PFS and PEA cases are shown in Figure 24.16. The four Kamoa North mines ranked by their relative values are:

- Kamoa Ouest Mine 1 (PEA 6 Mtpa).
- Kansoko Nord Mine 2 (PEA 6 Mtpa).
- Kamoa Centrale Mine 3 (PEA 6 Mtpa).
- Kamoa Nord Mine 4 (PEA 3 Mtpa).

Figure 24.16 Kamoa-Kakula IDP19 Mining Locations

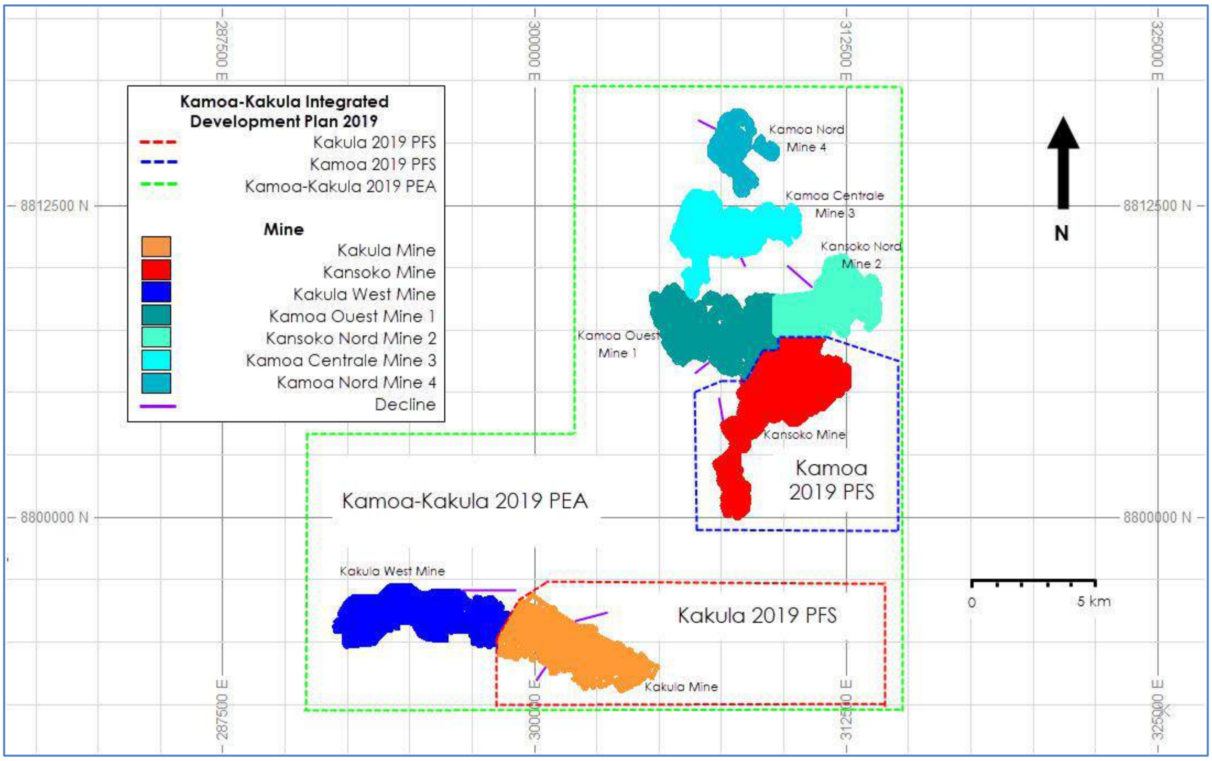


Figure by OreWin, 2019.

The Kamoa North mines are based on a preliminary UG optimisation at $80/t NSR cut-off grade. The Resources model name is kam14a160309.dm.

Stope optimisation was undertaken on the resource model at mining cut-off grades of $80/t NSR, then the outlier blocks that did not provide a consistent mineable shape were removed from the targeted resource. The Kamoa North targeted resource has been divided to five separate mines based on the shape of the minable material. Mine 5 has not been included in the PEA study due to the small amount of material. Figure 24.17 shows the optimised block by Kamoa North mines.

Figure 24.17 Kamoa Stope Optimisation by Mines

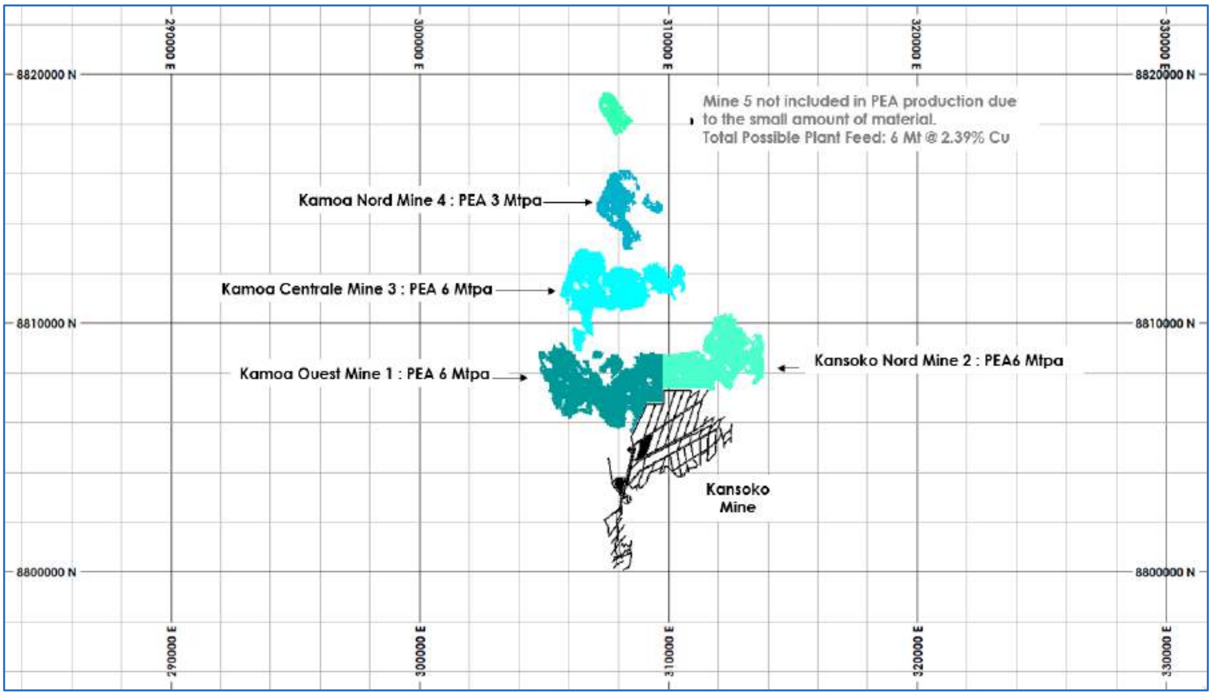


Figure by OreWin, 2019.

The proposed mining methods for Kamoa North Mine were assumed to be a combination of controlled convergence room-and-pillar mining method and room-and-pillar mining method based on the Kamoa 2019 PFS study.

Two mining methods were chosen:

- Controlled convergence room-and-pillar.
- Room-and-pillar.

The criteria to select the mining methods for the stope were the height and the dip of the optimised stope. The controlled convergence room-and-pillar method was selected for heights greater than 3 m and less than 6 m and dip less than 25°. The room-and-pillar was selected for heights greater than 6 m. The room-and-pillar method was also selected for heights greater than 3 m and less than 6 m and dip greater than 25° as follow:

- Controlled convergence room-and-pillar: Optimised Dip <= 25.0°.
- Room-and-pillar: Optimised Dip > 25.0°.

The Kamoa North preliminary optimisation was completed in four main phases as follows:

- MSO optimisation to identify the potential minable inventories and area.
- Identify the consistent mineable shapes and remove the optimised stopes that did not create a consistent mineable shape.
- Adjust the mining methods for the stopes that provide a small minable area. The mining method of small mining zones were changed to be the same as the surrounding mining method. The room-and-pillar mining method was also changed to controlled convergence room-and-pillar method where the dip of the stope is less than 25°. In this case, the height of the new controlled convergence room-and-pillar stopes were changed to 6.00 m and then the mined tonnes were adjusted based on the new stope height.
- Split the Kamoa North Mining area into five separate mines base on a consistent mine shape and the dimension of each mine.

Access will be via the twin declines to the mining zones of the Kamoa North mines. Selection of the mining method was dictated by mining height and dip and the size of the mining zones. One of the declines will be the main service access to the underground mine and the conveyor haulage system will be installed in the other decline. The material mined will be hauled to the transfer points and then transported by conveyor to surface.

The mine lateral and vertical development design has not been done for Kamoa North mines in 2019 PEA study. The lateral and vertical development and the other underground development requirement have been factored from Kakula West 2019 PEA development for mine production schedule and the financial model purpose. The metres of the required lateral and vertical development to access the mining zones were estimated for each mine separately as shown in Table 24.14.

A series of the ventilation shafts and raise will be developed to provide the fresh air for the production panels and development area. The ventilation shaft in Kamoa Ouest Mine 1 and Kansoko Nord Mine 2 will decrease the mine production ramp up period and increase the production to feed the plant as soon as possible. Mining costs were developed using the contractor mining costs from the current development at the Kansoko Mine, and factored fixed costs and unit rates from the Kakula 2019 PFS and the Kamoa 2019 PFS.

The Kamoa North optimisation, development and production need to be further investigated as a part of future studies.

Table 24.14 Kamoa North Mines Development Required

	Unit	Kamoa Ouest Mine 1	Kansoko Nord Mine 2	Kamoa Centrale Mine 3	Kamoa Nord Mine 4
Mine Production Rate	Mtpa	6 Mtpa	6 Mtpa	6 Mtpa	3 Mtpa
Lateral Development					
Conveyor Decline	m	731	1,791	232	768
Service Decline	m	731	1,791	232	768
Lateral Development		55,582	64,257	42,828	15,093
Conveyor Drift	m	5,500	3,800	5,700	3,600
Total Lateral Development	**m**	**62,543**	**71,640**	**48,993**	**20,229**
Vertical Development					
Ventilation Shaft	m	70	200	–	–
Ventilation Raise	m	1,280	4,110	1,070	450
Total Vertical Development	**m**	**1,350**	**4,310**	**1,070**	**450**

24.6 Kamoa-Kakula 2019 PEA Processing and Infrastructure

The Kamoa-Kakula 2019 PEA scenario assumes that the project proceeds with first completing the Kakula 6 Mtpa process plant, built in two stages of 2 x 3 Mtpa, followed by an expansion to include the 6 Mtpa Kamoa concentrator plant to take production to 12 Mtpa. A final expansion of a third 6 Mtpa concentrator stream is included to take the central complex's processing capacity to 18 Mtpa. The Kamoa-Kakula 2019 PEA processing and infrastructure facilities include:

- A 18 Mtpa central processing facility complete with surface crushing and screening, milling, and flotation, consisting of three 6 Mtpa concentrator streams, a smelter, and associated infrastructure located at the Kakula Mine area.
- The Kakula Mine and dedicated surface infrastructure on the Kakula Deposit.
- The Kansoko Mine on the Kansoko Sud and Kansoko Centrale areas of the Kamoa Deposit; including associated overland conveying systems.
- Dedicated surface infrastructure including associated overland conveying systemsat at Kakula West Mine and Kamoa North Mines 1 to 4.

The annual production results including processing, concentrate and smelter production are shown in Table 24.9 to Table 24.11.

24.6.1 Central Processing Facility

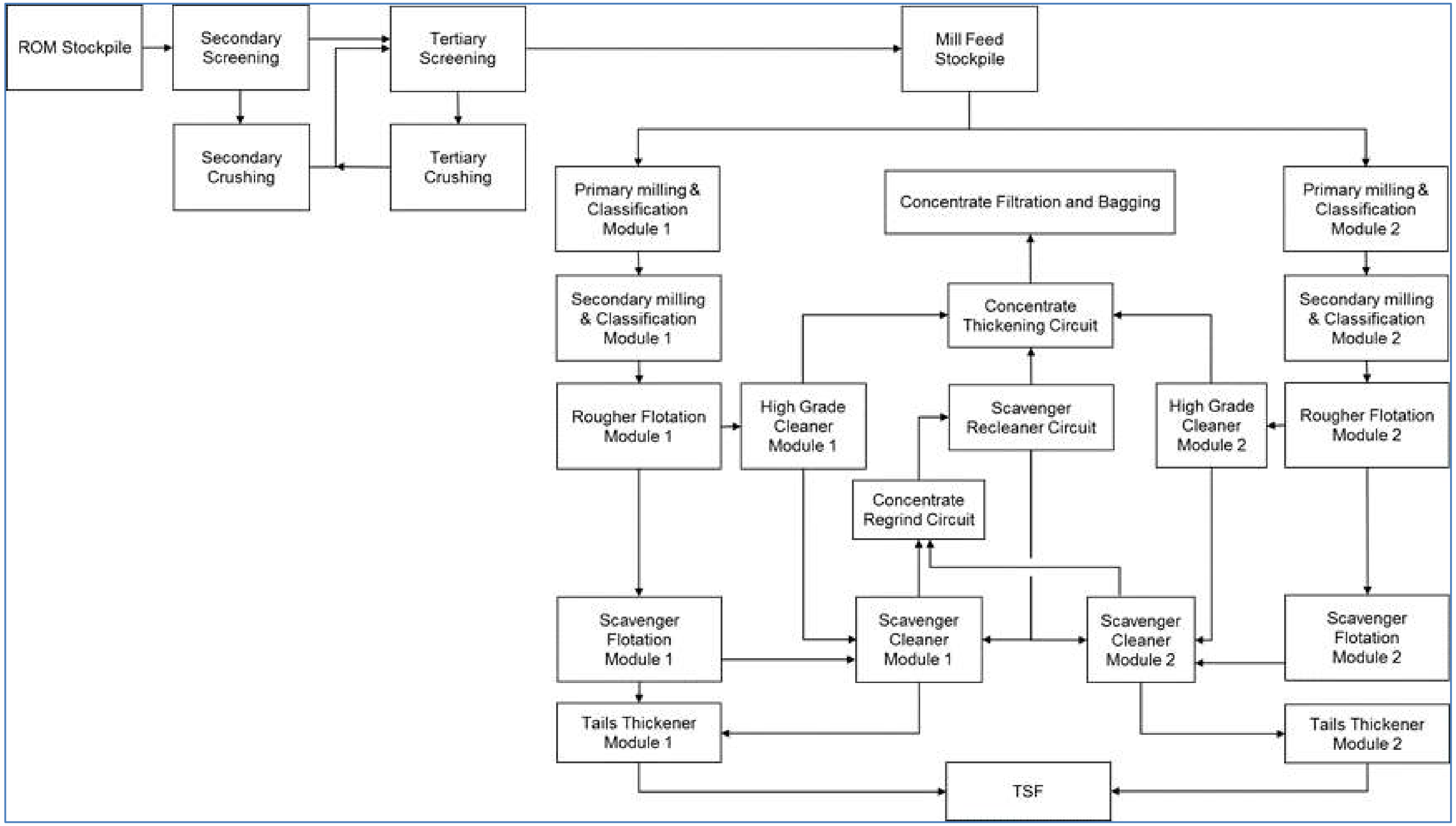
The Kakula process plant will be the first of three 6 Mtpa circuits to be located at the central processing complex. The Kakula concentrator (Central Complex Concentrator 1) includes a 15,000 t ROM stockpile to feed a 6 Mtpa Run-of-Mine (ROM) concentrator based on staged crushing and screening, followed by two stage, series, ball milling. The ball milling product is upgraded in the flotation circuit which is designed to produce two different concentrate product, i.e. a high grade and a medium grade product. These two concentrate products are combined to form the final concentrate.

The Kakula design allows the Central Complex Concentrator 1to be built into two phases in order to be aligned with the mine production schedule. Phase 1 will treat 3 Mtpa in line with the mine ramp up and the throughput will be doubled during Phase 2 to 6 Mtpa. Refer to Section 17 for more detail on the Kakula design. A high level block flow diagram of the Central Complex Concentrator 1 is presented in Figure 24.18.

Following the ramp-up of Central Complex Concentrator 1 to 6 Mtpa, the complex will be expanded by the addition of the Central Complex Concentrator 2 at the Kakula Mine Area. Central Complex Concentrator 2 will be based on the Kansoko circuit design. The expansion from 6 Mtpa to 12 Mtpa will also be completed in a two phased approach, as dictated by the mining plan.

Kansoko ROM material will be stored on the 15,000t Kansoko Mine stockpile (located at the Kansoko Mining Area) from where it will be extracted by a duty/standby apron feeder arrangement, prior to discharging onto the 6km Kansoko Overland Conveyor 1. The Kansoko Overland Conveyor 1 in turn will discharge the material onto the 11km Kansoko Overland Conveyor 2, which will transfer the Kansoko material to the Central Complex Concentrator 2 ROM stockpile (15,000t). Central Complex Concentrator 2 also consists of a 6 Mtpa Run-of-Mine (ROM) concentrator based on staged crushing and screening, followed by two stage, series, ball milling. The ball milling product is upgraded in the flotation circuit which is designed to produce two different concentrate product, i.e. a high grade and a medium grade product. These two concentrate products are combined to form the final concentrate. There are minor variations in the design of Concentrator 1 and Concentrator 2, as each flow sheet was tailored with a specific orebody in mind. Refer to Section 17 for more detail on the Central Complex Concentrator 2 design.

Figure 24.18 Central Complex Concentrator 1 Block Flow Diagram



A high level block flow diagram of the Central Complex Concentrator 2 circuit is shown in Figure 24.19. Once Central Complex Concentrator 2 has ramped up to 6 Mtpa, the complex will be expanded by the addition of the Central Complex Concentrator 3 at the Kakula Mine Area. Central Complex Concentrator 3 will also be based on the Kansoko circuit design (Figure 24.19). The mine plan again requires that the expansion from 12 Mtpa to 18 Mtpa be completed in a two phased approach.

Figure 24.19 Central Complex Concentrator 2, 3 Block Flow Diagram

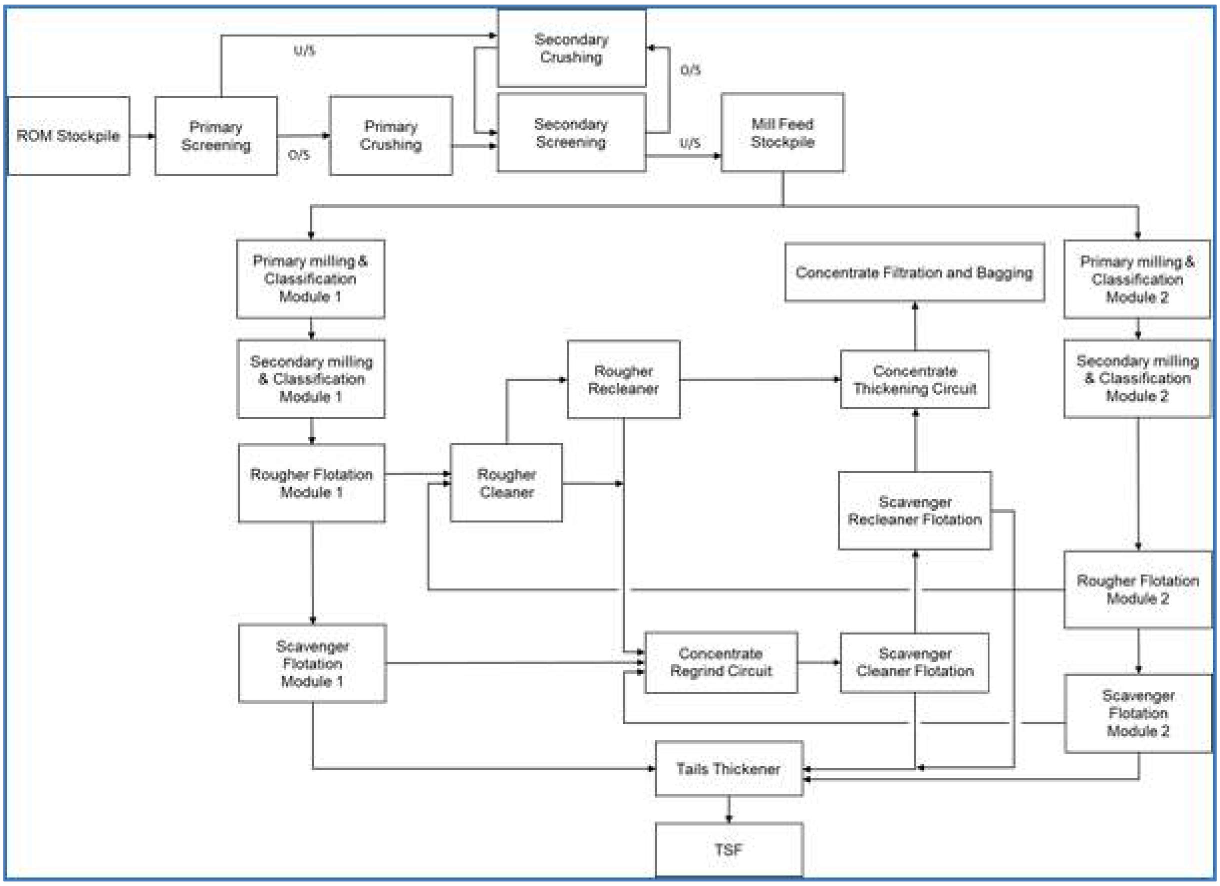


Figure by DRA, 2018

Kakula West ROM material will be stored on the 15,000 t Kakula West Mine stockpile (located at the Kakula West Mining Area) from where it will be extracted by a duty/standby apron feeder arrangement, prior to discharge onto the 3 km Kakula West Overland Conveyor to transfer the Kakula West material to the Central Complex Concentrator 3 ROM stockpile (15,000 t).

After approximately 17 years of mining, at the Kakula Mine, the output of the mine starts to deplete, and The Kamoa North Mine 1 is brought on line to supplement tonnage to the central complex. Kamoa North Mine 1 ROM material will be stored on the 15,000 t Kamoa North Mine 1 stockpile (located at the Kamoa North Mine 1) from where it will be extracted by a duty/standby apron feeder arrangement, prior to discharge onto the 2 km Kamoa North 1 Overland Conveyor. The Kamoa North 1 Overland Conveyor in turn will discharge the material onto the 11 km Kansoko Overland Conveyor 2, which will transfer the Kamoa North 1 ROM material to the central processing facility. At the processing facility, the Kansoko Overland Conveyor 2 will discharge onto a transfer conveyors feeding the Central Complex Concentrator 1 ROM stockpile.

Once the Kakula material is mined out, and the Kakula West material starts to deplete, the Kamoa North 2 Mine and the Kamoa North 3 Mines are brought online to maintain the 18 Mtpa processing rate.

Kamoa North Mine 2 ROM material will be stored on the 15,000 t Kamoa North Mine 2 stockpile (located at the Kamoa North Mine 2) from where it will be extracted by a duty/standby apron feeder arrangement, prior to discharge onto the 6 km Kamoa North 2 Overland Conveyor.

Kamoa North Mine 3 ROM material will be stored on the 15,000 t Kamoa North Mine 3 stockpile (located at the Kamoa North Mine 3) from where it will be extracted by a duty/standby apron feeder arrangement, prior to discharge onto the 6 km Kamoa North 3 Overland Conveyor.

Both the Kamoa North 2 Overland Conveyor and the Kamoa North 3 Overland conveyor will discharge material onto the 11 km Kansoko Overland Conveyor 2 for transfer to the central processing facility. At the processing facility, the Kansoko Overland Conveyor 2 will discharge onto various transfer conveyors to discharge the material onto either one of the three ROM stockpiles (Concentrator 1, 2, and/or 3) in order to maintains stockpile levels between the various circuits.

Towards the end of LOM, the Kakula West and Kansoko material is mined out, and the Kamoa North 4 Mine is brought into production to maintain the 18 Mtpa processing rate. Kamoa North Mine 4 ROM material will be stored on the 15,000 t Kamoa North Mine 4 stockpile (located at the Kamoa North Mine 4) from where it will be extracted by a duty/standby apron feeder arrangement, prior to discharge onto the 6 km Kamoa North 4 Overland Conveyor. The Kamoa North 4 Overland Conveyor will discharge material onto the 11 km Kansoko Overland Conveyor 2 for transfer to the central processing facility.

24.6.2 Kamoa-Kakula 2019 PEA Smelter

Concentrate will be conveyed from the adjacent concentrator complex into a concentrate shed located in the smelter complex. The smelting process utilizes the direct-to-blister smelting technology(DBF), which is proven for treating high copper, low sulphur copper concentrates similar to those envisaged for the Kamoa-Kakula project. Copper concentrate is fisrt dried in a steam dryer before being oxized with oxygen enriced air in the reaction shaft of the DBF to produce blister copper and SO_2 offgas in a single stage flash smelting process. The SO_2 laden offgases are dedusted and sent to a double-contact-double adsorption sulphuric acid plant for production of high strength acid, which is sold to the local market. Copper is recovered from the DBF slag in a downstream electric slag cleaning via reduction with metallurgical coke to produced blister copper. Electric furnace slag still containing up to 4% copper, is slowed cooled, crushed and sent to a slag flotation plant to receovery the residual copper, in the form of concentrate, which is then back to the coneontrate storage shed and mixed with fresh concentrate. The final tailings from the slag plant containing 0.8% copper will be pumped to the central complex concentrator tailings facility.

The smelter is designed to process 1,000 ktpa concentrate, producing up to 467 ktpa blister copper and an average 710 ktpa of high strength sulphuric acid. The flow sheet schematic of the DBF process is shown in Figure 24.20. The smelter feed copper-to-sulphur ratio is maintained above 1.4 for energy balance purposes. In the latter years where the ratio drops below 1.4, the feed rate is throttled to maintain the energy balance. Excess concentrate will be sold to the market.

Figure 24.20 Flow Sheet Schematic of the Direct-to-Blister Smelting Process

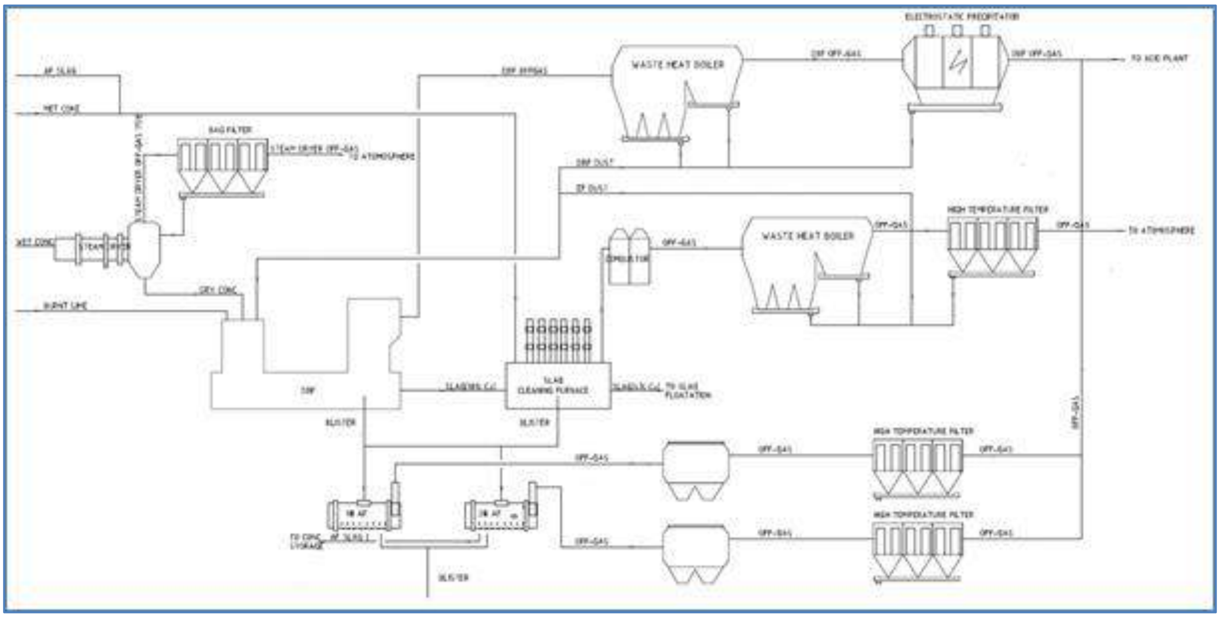


Figure by Nerin, 2018.

24.6.3 Kamoa-Kakula 2019 PEA Infrastructure

The infrastructure for the Kamoa-Kakula 18Mtpa PEA must support three, 6Mtpa concentrator circuits, at a centralised processing facility, as well as dedicated mine surface infrastructure at each of the individual mine sites.

The project infrastructure includes power supply, tailings dams, communications, logistics, transport options, materials handling, water and waste water, buildings, accommodations, security, and medical services.

24.6.3.1 Power

Power for the Kamoa-Kakula Project is planned to be sourced from the DRC's state-owned power company (SNEL, Société Nationale d'Electricité) electrical interconnected grid. Kamoa has secured sufficient power for the initial stages of the project (as described in Section 18). Kamoa is working very closely with SNEL at securing additional power for the Project's long term plans. Bulk power suplly will be ramped up as needed to meet the production requirements.

24.6.3.2 TSF

The Kamoa-Kakula 18 Mtpa PEA considered the expansion of the Kakula TSF to allow for the additional tailings tonnage. Refer to Section 18 for a detailed description of the Kakula TSF.

24.6.3.3 Site Access and Transport

The main access road to the Kakula site is currently being constructed as part of the Kakula project to allow for future mining activities required for the Kamoa-Kakula Copper Project. This road gives access from Kolwezi to Kakula mine and is divided into two sections:

- Section 1: Section from Kolwezi Mines turnoff to Kansoko.
- Section 2: Kansoko Mine to Kakula Mine.

The internal road design philosophy is that delivery vehicles, LDVs and concentrator trucks will remain on separate roads to the required delivery points, working and parking areas. With internal roads reserved for the delivery of equipment from stores to the applicable work area. The internal roads and parking take account of the traffic flow inside the mine area. Security gates separate areas in order to control access, without reducing serviceability and production.

Different types of surface finish and layer works have been designed for differing types of application and road uses within the mine area. The following have been used as part of the mine design and layout:

- Asphalt road layer works used for the main entrance access road,
- Paving road layer works where heavy equipment and trucks are turning, and
- Gravel road layer works used mainly in the outer portions of the concentrator plant and site infrastructure for maintenance access.

24.6.3.4 Water Supply

Raw water will be provided to the site via production boreholes, mine dewatering boreholes and mine decline dewatering. This will provide all necessary raw water which will then be used to provide the required process water makeup, gland water, fire and reagent make-up water. Return water pipelines will bring water from the TSF to the associated process water tanks for re-use.

Due to the high annual rainfall, local dams and rivers and mine dewatering, ample water is available to satisfy the required water demand for both plants. It is envisaged that all raw water can be supplied from the available ground water sources.

Potable water for local villages is currently obtained from local rivers and streams. Potable water for any future mining operation will be sourced from boreholes. Potable water for ablution facilities, kitchens and emergency stations (eyewash and showers) will be obtained from the bulk water system and treated by means of disinfection only (chlorination). An appropriate drinking water standard will be applied, referencing indicators such as bacterial content, residual chlorine, turbidity, and dissolved solids.

24.6.3.5 Stormwater and Wastewater

The storm water management system will consists of storm water run-off drains, storm water dams, and a discharge drain(s). The storm water run-off drains are a network of drains running through the mining area collecting all run-off water and directing it towards appropriate storm water ponds. These drains vary in size and all are concrete lined.

Discharging of the collected clean water into the nearest river, will be via a discharge drain, designed to minimise potential flooding of surroundings.

Dirty water collected in storm water dams will discharge for events over and above 1:50 to the nearest watercourse.

24.6.3.6 General Infrastructure

Fuelling infrastructure has been allowed for at the central processing facility to cater for the concentrators, while dedicated fuelling infrastructure has been included at each of the various mine sites.

On site workshops have been allowed for at each of the various mine sites, as well as the central processing facility, to facilitate repair the mobile machinery on site. If vehicles break down on route to site, commercially-owned breakdown rigs with a towing capacity of up to 30 t are available.

Within the infrastructure costing, allowance has been made for camp, together with plant and perimeter security fencing. The fence follows a maintenance laterite access road providing patrol and fence maintenance access. Security control buildings at major access control points have been allowed for, including ablution facilities.

The fire protection and detection systems for the surface plant and infrastructure (excluding all underground mining which is covered separately) will be developed in consultation with, and subject to final approval from, the Owner's risk assessors. The system will be designed to comply with DRC legislation, the project Health and Safety standard/s, project specifications and fire protection standards as adopted by the Project.

The clinic and first-aid facility will be housed together at a suitable position near the main gate. Medical equipment, including an ambulance, will be provided. Medical evacuation for ex-patriate employees will be provided by an outside contracting service.

Permanent villages called the Owners Camps, capable of accommodating 1500 persons each, will be constructed at the mine locations to provide accommodation for owner's team management, expatriates and consultants. Single units will comprise of one and two bed shared ablution facilities and family units with two bedrooms and bathroom with open plan living room and kitchen.

The Owners Camp will be constructed upfront and utilised as the project construction camp. The camp will accommodate the construction workers during execution and will be erected within walking distance of the operations.

An integrated approach to waste management for the Kamoa-Kakula Project will be required. This would involve reduction, reuse, recycling and would be done onsite through waste separation. A non-hazardous landfill site is planned at the Kamoa-Kakula Project.

24.6.3.7 Construction Facilities

To facilitate the execution of the project, various temporary facilities need to be put in place. These facilities include:

- Construction Site Offices: The Mine Services Building will be constructed upfront to accommodate the client site team as well as the EPCM consultants. These offices will include ablutions and conference rooms and will have facilities to communicate with head offices and receive and print construction drawings.
- Laydown areas: Contractors will require prepared areas to establish their site offices and areas to store construction material, equipment and vehicles. Fenced terrace areas with water, sewer and temporary electrical connections will be provided.
- Customs Clearance Area (Bonded Area): To facilitate the smooth delivery and release of construction material ordered from outside the DRC, a customs clearance area will be created on site from which a customs clearance official will check, register and release all imported construction material. Fenced terrace areas with office, small store, water, sewer and electrical connections will be provided.

Earthworks shall be designed with suitable grading for quick elimination of surface run-off and keeping in mind optimisation of cut-and-fill earthworks quantities. Stepped terraces shall be proposed to accommodate mechanical and process requirements on the plant.

24.6.4 Kamoa-Kakula 2019 PEA Process and Infrastructure Costs

The Kamoa-Kakula 18 Mtpa PEA analysed a six-phased expansion of production from 3 Mtpa to 18 Mtpa (in 3 Mtpa increments). The following inputs and documents were identified and used in compiling the capital cost estimate:

- Kamoa PFS capital cost estimate (as detailed in Section 21),
- Kakula PFS capital cost estimate (as detailed in Section 21), and
- China Nerin Engineering Co. Ltd smelter study capital cost estimate.

Costs have been estimated for the following disciplines:

- Earthworks.
- Civil works.
- Structural steel fabrication, supply and erection.
- Platework fabrication, supply and erection.
- Mechanical equipment supply and installation.
- Pipework fabrication, supply and erection.
- Electrical and C&I supply and erection.
- Transportation to site.
- EPCM services.
- First fills of consumables.
- Spares.
- Infrastructure.

The operating cost estimate includes the fixed (labour and maintenance) costs and variable costs components (reagents, grinding media and power costs). The operating cost figure excludes rehabilitation, mining, insurance costs, import duties and all other taxes.

24.6.5 Comments on Section 24.6

ROM from satellite deposits (Kansoko, Kakula West, and Kamoa North) is assumed to have a top size of 350 mm; however, a top size of 250 mm will be more conducive to overland conveying. It is noted that testwork has been conducted to determine the flotation performance of Kakula West and Kansoko material in the Kakula flow sheet. Both deposits showed a favourable response to the Kakula flow sheet. Limited samples from the Kamoa North Mine area was subjected to flotation testing during the earlier Kamoa testwork campaings. It is recommended that the flotation response of the Kamoa North deposits be tested on the Kakula PFS and Kansoko flow sheets.

During the infrastructure planning for the Kakula 18 Mtpa PEA no issues were identified that may have a material negative impact on the financial viability of the project. Synergy with regards to shared infrastructure, with possible resultant cost reductions, will be reviewed between the various mine sites infrastructure during the next stage of the study.

24.7 Kamoa-Kakula 2019 PEA G&A and Owners Costs

Owners and General and Administration (G&A) costs were developed using factored fixed costs and unit rates from the Kakula 2019 PFS and the Kamoa 2019 PFS. Allowances for operating the centralised process complex at Kakula including the concentrator and smelter were included. The G&A requirements for the multiple mines were based on review of the production schedule an the number and location of the mines as they were developed, brought into production and completed.

25 INTERPRETATION AND CONCLUSIONS

25.1 Mineral Resource Estimate

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). Amec Foster Wheeler has checked the data used to construct the resource model, the methodology used to construct them (Datamine macros) and has validated the resource model. Amec Foster Wheeler finds the Kamoa resource model to be suitable to support prefeasibility level mine planning, and the Kakula resource model is suitable to support prefeasibility level mine planning.

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

- Drill spacing.
    - The drill spacing at the Kamoa and Kakula deposits is insufficient to determine the effects of local faulting on lithology and grade continuity assumptions. Local faulting could disrupt the productivity of a highly-mechanised operation. In addition, the amount of contact dilution related to local undulations in the SMZ has yet to be determined for both deposits. Ivanhoe plans to study these risks with the declines currently in progress at Kamoa and Kakula.
    - Delineation drill programs at the Kamoa deposit will have to use a tight (approximately 50 m) spacing to define the boundaries of mosaic pieces (areas of similar stratigraphic position of SMZs) in order that mine planning can identify and deal with these discontinuities. Mineralisation at Kakula appears to be more continuous compared to Kamoa.
    - Assumptions used to generate the data for consideration of reasonable prospects of eventual economic extraction for the Kamoa deposit.
    - Mining recovery could be lower and dilution increased where the dip locally increases on the flanks of the domes. The exploration decline should provide an appropriate trial of the conceptual room-and-pillar mining method on the Kamoa deposit in terms of costs, dilution, and mining recovery. The decline will also provide access to data and metallurgical samples at a bulk scale that cannot be collected at the scale of a drill sample.
    - Assumptions used to generate the data for consideration of reasonable prospects of eventual economic extraction for the Kakula deposit.
    - A controlled convergence room-and-pillar technique is being studied which provides the opportunity for reduced costs.
- Metallurgical Recovery Assumptions at Kamoa.
    - Metallurgical testwork at the Kamoa deposit indicates the need for multiple grinding and flotation steps. Variability testwork has been conducted on only portions of the Kamoa deposit. Additional variability testing is needed to build models relating copper mineralogy to concentrate grade and improve the recovery modelling.
    - A basic model predicting copper recovery from certain supergene mineralisation types has been developed. More variability testing is required to improve this model to the point where it is useful for production planning purposes.

- Metallurgical Recovery Assumptions at Kakula.
  - Preliminary metallurgical testwork at the Kakula deposit indicates that a high-grade chalcocite-dominant concentrate could be produced at similar or higher recoveries compared to those achieved for Kamoa samples.
  - There is no supergene mineralisation currently identified at Kakula that requires a dedicated recovery model separate from the hypogene recovery prediction method.
  - Exploitation of the Kamoa-Kakula Project requires building a greenfields project with attendant 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 Kamoa and Kakula Mineral Resource estimates.
- Commodity prices and exchange rates.
- Cut-off grades.

25.2 Kamoa-Kakula Integrated Development Plan 2019

The development of Kamoa-Kakula should be reassessed for the impact of the Kakula West Mineral Resource in order to determine the relative value of Kakula West against the other areas within the Kakula and Kamoa Mineral Resources. The Kamoa-Kakula IDP19 includes an update of the Kamoa Mineral Reserve and updates of the preliminary economic assessment (PEA) on the Kakula Mineral Resource.

The analysis in the Kamoa-Kakula 2019 PEA indicates that discovery of the Kakula deposit has changed the potential development scenarios for the Kamoa-Kakula project, and additional studies should be prepared to define the development sequence and production rates including mining methods, plant sizing and location for the deposits.

The Kamoa-Kakula IDP19 is an update of the Kakula Mineral Resource including Kakula West. The development scenario should be tested and reviewed to determine updates to the development plan.

25.3 Mineral Reserve Estimate

25.3.1 Kakula Mineral Reserve Estimate

Mineral Reserves for the Kakula 2019 PFS conform to the requirements of CIM Definition Standards (2014). Stantec has utilised development processes and cost estimates to the level of accuracy required to state reserves and support a prefeasibility-level study. Areas of uncertainty that may impact the Mineral Reserve Estimate include:

- Commodity prices and exchange rates.
- The continuity and dip of the ore will need to be better defined prior to and during the mining stages.
- The security and timeliness of binder supply to the paste plant at Kakula.
- The amount of groundwater present in the orebody during the mining cycle.

25.3.2 Kamoa Mineral Reserve Estimate

Mineral Reserves for the Kamoa 2019 PFS conform to the requirements of CIM Definition Standards (2014). Stantec has utilised development processes and cost estimates to the level of accuracy required to state reserves and support a prefeasibility-level study. Areas of uncertainty that may impact the Mineral Reserve Estimate include:

- Commodity prices and exchange rates.
- Ground reaction to the controlled convergence room-and-pillar mining method. To address this, the schedule allows for a trial panel to be 80% extracted prior to beginning other controlled convergence room-and-pillar areas.
- The continuity and dip of the ore will need to be better defined prior to and during the mining stages.

25.4 Metallurgy and Process Plant

It is the opinion of the qualified person that acceptable metallurgical testwork programmes were conducted, on representative samples of the Kamoa and Kakula deposits. Observations and conclusions from the testwork were accurately interpreted and included in the respective PFS concentrator designs. Further variability testwork on the Kamoa deposit is required to test the robustness of the Kamoa concentrator design, while a high level of robustness of the Kakula concentrator circuit design has been illustrated by the preliminary variability testwork conducted on the Kakula material as part of the Kakula 2019 PFS.

25.5 Infrastructure

General surface infrastructure addressed power supply from source to mine in sufficient detail. Agreements between Kamoa and SNEL are in place to secure power for the mine.

Material supply to site is contracted out to a well-established transportation company with transportation experience to site and into the DRC. This will ensure material and equipment reach operations within a reasonable time period.

Various transport methods and routes between rail, road and ocean transport has been considered and identified by an experienced logistics consultanr, for the transport of concentrate out of country.

The Kamoa and Kakula Projects are reliant on boreholes for the supply of construction water during the first two years, after which the project has excess water that needs to be discharged to the environment. Discharge to the environment of treated water occurs from either the Water Treatment Plant or the Sewerage Treatment Plant.

26 RECOMMENDATIONS

26.1 Further Assessment

Ivanhoe now has three areas within the Kamoa-Kakula Project (Kamoa, Kakula and Kakula West) that warrant further assessment and are at different stages of study and development. Kakula is a very high-grade Mineral Resource that is separate to Kamoa and could be developed as a separate mine and processing facility, and given this, further study should be undertaken. The Kamoa-Kakula 2019 PEA has identified potential development scenarios for Kamoa and Kakula deposits that suggest expansion of the initial project. The next phase of detailed study should be to prepare a feasibility study on Kakula. A whole of project approach should be undertaken to optimise the project and to take the project through the study phases to production. The key areas for further studies are:

- The Kakula 2019 PFS has identified a Mineral Reserve and development path for the Kakula Deposit. It is recommended that Kakula be progressed to a feasibility study.
- The Kamoa-Kakula 2019 PEA indicates that there is potential value in a central processing facility, on-site smelting and expansions in production. In order to identify this potential, further study will be needed. It is recommended that these studies are undertaken using a whole of project approach into the long-term options to maximise the efficient extraction of the Kamoa-Kakula Mineral Resources.
- Rail and power options for the Project remain important considerations and studies to increase the confidence in the assumptions should continue.
- Continue to monitor the regulatory provisions to be adopted, ensuring as far as possible, continued adequate adherance to the relevant legislative requirements.
- Revisions and updates of the long-term whole of project planning as the Mineral Resources are further defined. Including expanding and optimising the project production rate by considering concentrator and smelter capacities that are matched to the power supply availability, mine production and transport options.
- Other mining areas and additional mines from the Kamoa deposit.
- Rail transport to Lobito.
- Continue infill drilling programme to upgrade resource categorisation, enhance geotechnical database and its application to mine design and ground support, and better understand the continuity of the deposit and impacts on productivities and dilution.
- Consider an underground exploration programme at Kakula to attain first-hand information on actual mining conditions and to validate design assumptions.
- Complete hydrological studies and data evaluation to better determine impacts on underground mining conditions and productivities.

26.2 Drilling

Extensive drilling has been completed at Kakula and Kakula West, and the goal of establishing sufficient Indicated Mineral Resources to support stand-alone mining operations at both Kakula and Kakula West has been achieved. The future drill plan at Kakula is to scale down the drilling significantly to only those drillholes required to define the edges of the higher-grade material, and continue infill drilling at Kakula West. While exploration drilling will continue, the drilling will focus on targets elsewhere within the Project and continue at Kamoa North to better define the recently-discovered high-grade corridors. The drill plan is expected to adjust as ongoing results become available. Amec Foster Wheeler has recommended a total 2019 programme of 55,000 m at a cost of $9.0M.

26.3 Underground Mining

The following is a list of mining recommendations for the Project:

- Monitor the initial mining block at Kakula to attain first-hand information on actual mining conditions and to validate the design assumptions.
- Monitor the initial panel of the controlled convergence room-and-pillar mining at Kamoa to attain first-hand information on actual mining conditions and to validate design assumptions.
- Continue the infill drilling programme to upgrade resource categorisation, enhance geotechnical database and application to mine design and ground support, and better understand the continuity of the deposit and impacts on productivities, recoveries, and dilution.
- Consider an underground exploration programme at Kamoa to attain first-hand information on actual mining conditions and to validate design assumptions.
- Complete hydrological studies and data evaluation to better determine impacts on underground mining conditions and productivities at Kamoa.
- Drill geotechnical holes to determine ground conditions at each ventilation raise.
- Monitor KPS zones for changing ground conditions and apply the findings.
- Determine the virgin rock temperature gradient.
- Develop an operating philosophy to optimise waste rock going into room-and-pillar goaf, and drift-and-fill areas.
- Evaluate the production impact of eliminating the low-profile equipment fleet and using the standardise fleet for room-and-pillar mining.
- 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.
- Evaluate the underground crushing requirements and location of such infrastructure at Kakula.

- Investigate the possibility of adding sand to the tailings at Kakula to reduce the binder quantity.
- Start the basic engineering for the first 5 years to reduce the chance of delays to the project schedule.

26.4 Process Plant

The following is a list of process recommendations for the Kamoa deposit:

- Kamoa Copper SA should develop a reliable and economic measurement method to determine the copper mineralogy of samples. This will be able to predict concentrate grades and copper recoveries. Planned variability testing must proceed and the suitability of the IFS4a flotation flow sheet must be critically analysed in light of the variability results.

  The most critical unresolved process issue is prediction of copper 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.
- A reliable prediction method is required for copper concentrate grade, based on either the Cu:S ratio or on measured copper mineralogy. A variability testwork program must be performed to establish, at a minimum, a useful predictive method.
- If a smelter is considered for future studies, then the concentrate grade prediction method requires a high level of accuracy when compared to a concentrate sales-based project. Incorporation of a smelter in a PFS will require a more extensive characterization and flotation variability testwork program compared to a PFS that excludes smelting.
- The value of using %ASCu in determining copper recovery from surface-oxidised supergene samples must be confirmed by a program of sample analysis and flotation variability testwork.
- The current method of predicting copper recovery using %ASCu, assuming it is proven useful, should be targeted for refinement in the variability flotation testwork program.
- The currently preferred ASCu determination method may be dissolving copper that is easily floatable (chalcocite and covellite) and alternative methods (weaker acid, alternative acids, etc) should be explored within the flotation variability testwork program.

The following is a list of process recommendations for the Kakula deposit:

- Locked cycle testwork to be conducted on the Kakula material and compared against the performance achieved in the open circuit batch testwork.
- Concentrate regrind testwork to be completed during the next project phase to validate current regrind circuit design parameters, which are based on the historical Kansoko results.
- Further variability testwork to be conducted to firm up on the current recovery model, as well as the inclusion of locked cycle test results in the recovery model.

The following is a list of process recommendations for the Kakula West deposit:

- Extensive testwork campaign to be initiated to determine recovery model of the Kakula West material on both the Kakula 2019 PFS and Kamoa 2019 PFS flow sheets.

The following is a list of process recommendations for the Kamoa North deposits:

- Extensive testwork campaign to be initiated to determine recovery model of the material from the various Kamoa North Mines, on both the Kakula 2019 PFS and Kamoa 2019 PFS flow sheets.

It is the opinion of the Process QP that the dominance of the hypogene and deep supergene ores in the project mean that the problems predicting supergene recoveries are not material to the Kamoa 2019 PFS. A lack of accurate prediction of copper concentrate quality from ore mineralogy could have material production effects in the scenario where a smelter is constructed as part of the project. However, sufficient time exists after commencement of the project to implement a high accuracy predictive method ahead of the currently envisaged smelter implementation. Lack of an accurate grade and quality prediction is not a material issue for concentrate sales scenarios, provided the customer's copper grade specification windows are reasonable.

26.5 Infrastructure

The Kansoko Mine in the Kamoa 2019 PFS was analysed without considering the Kakula Mine infrastructure and hence the Kamoa 2019 PFS should be updated to reflect infrastructure items that have been or will be moved to Kakula and vice-versa.

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