OreWin
Independent Mining Consultants

KAMOA COPPER SA

Kamoa-Kakula Project

Kamoa-Kakula Integrated Development Plan 2020

October 2020

Job No. 19013

IMPORTANT NOTICE

This notice is an integral component of the Kamoa-Kakula Integrated Development Plan 2020 (Kamoa-Kakula IDP20) and should be read in its entirety and must accompany every copy made of the Technical Report. The Kamoa-Kakula IDP20 has been prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects.

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Title Page

Project Name:	Kamoa-Kakula Project
Title:	Kamoa-Kakula Integrated Development Plan 2020
Location:	Lualaba Province Democratic Republic of the Congo
Effective Date of Technical Report:	13 October 2020
Effective Date of Mineral Resources:	Kamoa: 30 January 2020 Kakula: 10 February 2020
Effective Date of Drilling Database:	Kamoa Resource: 20 January 2020 Kakula Resource: 1 November 2018
Effective Date of Mineral Reserves:	8 September 2020

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Signature Page

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	Democratic Republic of the Congo
Effective Date of Technical Report:	13 October 2020

/s B F Peters

Date of Signing:

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/s G Seibel

Date of Signing:

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1.1 Introduction

The Kamoa-Kakula Integrated Development Plan 2020 (Kamoa-Kakula IDP20) 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). The Kamoa-Kakula Project (the Project) is in the Kolwezi District of Lualaba Province, Democratic Republic of Congo (DRC). The Project is held by Kamoa Copper SA (Kamoa Copper), a DRC company in which Ivanhoe has an indirect interest.

The Project is situated within the Central African Copperbelt in the DRC, approximately 25 km west of the provincial capital of Kolwezi, and about 270 km west of the regional centre of Lubumbashi.

The Project proposes underground mining of two extensive stratiform copper deposits called Kamoa and Kakula and processing the ore to produce a copper concentrate.

The previous Technical Report on the Project was the Kamoa-Kakula 2020 Resource Update with an effective date in March 2020.

1.2 Kamoa-Kakula Integrated Development Plan 2020

The Kamoa-Kakula IDP20 provides updates to the Project Mineral Reserves and the studies at Feasibility (FS), Prefeasibility (PFS) and Preliminary Economic Assessment (PEA) stages. The following are the key features of the Kamoa-Kakula IDP20:

- Kakula 2020 FS (Mineral Reserve with a plant throughput rate of 6.0 Mtpa).
- Kakula-Kansoko 2020 PFS (Mineral Reserve with a plant throughput rate of 7.6 Mtpa).
- Kamoa-Kakula 2020 PEA (Plant expansion to 19.0 Mtpa, smelter, Kakula, Kansoko, and six additional mines).

An overview of deposits included within the Kakula 2020 FS (outlined by blue dotted line), Kakula-Kansoko 2020 PFS (outlined by purple dotted line) and Kamoa-Kakula 2020 PEA (outlined by green dotted line) is shown in Figure 1.1.

Kakula and Kansoko are separate underground mines at Feasibility and Prefeasibility level of development respectively. The Kakula 2020 FS and Kakula-Kansoko 2020 PFS study include separate capital and operating costs and assumptions for underground mining, processing plant and infrastructure.

The Kamoa-Kakula 2020 PEA analyses a production case with an expansion of the Kakula concentrator processing facilities, and associated infrastructure to 19.0 Mtpa and includes a smelter and eight separate underground mining operations with associated capital and operating costs. The details of the Kamoa-Kakula 2020 PEA are provided in Section 24. The eight mines ranked by their relative net present values are:

- Kakula Mine (FS 6.0 Mtpa).
- Kansoko Mine (PFS 1.6 Mtpa to 6.0 Mtpa).
- Kakula West Mine (PEA 6.0 Mtpa).
- Kamoa North Mine 1 (PEA 6.0 Mtpa).
- Kamoa North Mine 2 (PEA 6.0 Mtpa).
- Kamoa North Mine 3 (PEA 6.0 Mtpa).
- Kamoa North Mine 4 (PEA 3.0 Mtpa).
- Kamoa North Mine 5 (PEA 1.0 Mtpa).

Figure 1.1 Kamoa-Kakula IDP20 Mining Locations

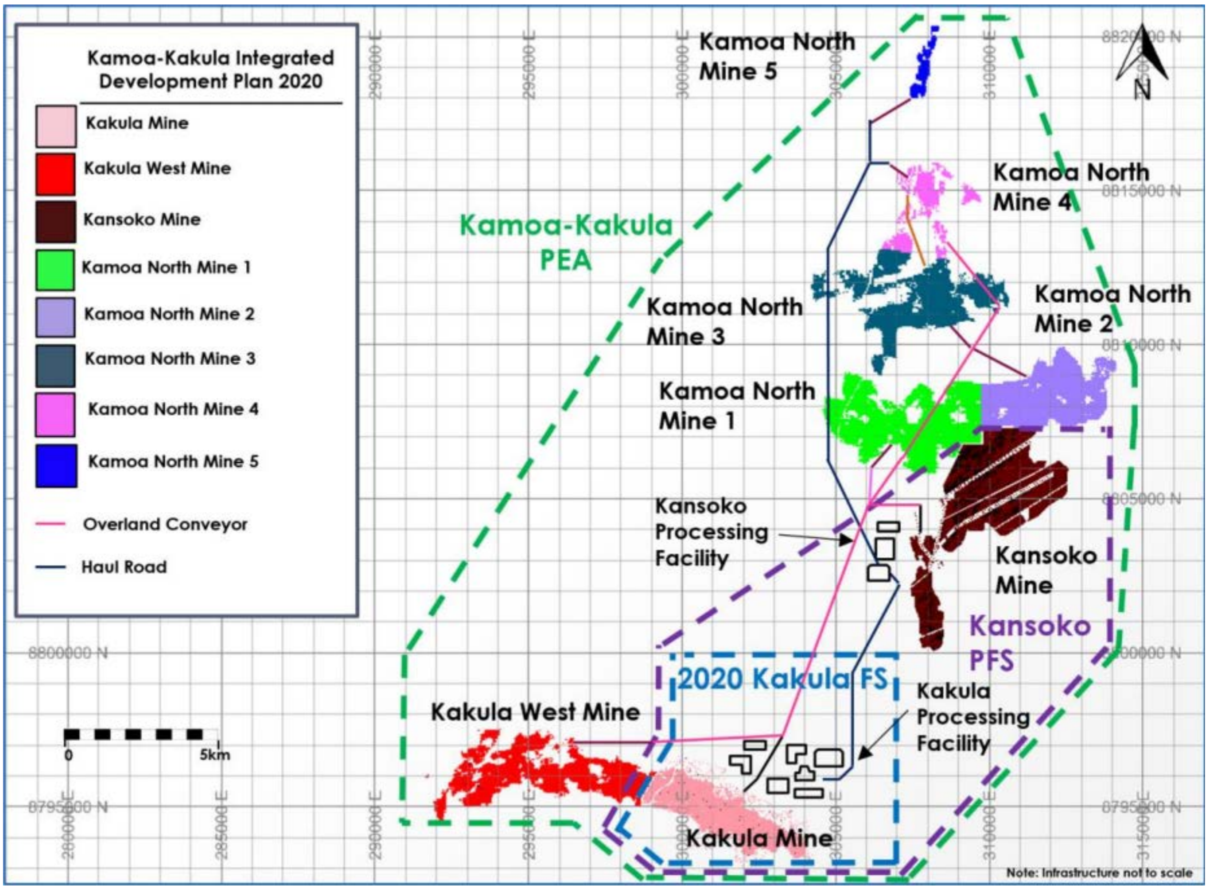


Figure by OreWin Pty Ltd, 2020.

The Kamoa-Kakula 2020 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 2020 PEA includes a PEA level study of the whole project including Kakula West and Kamoa North. Kakula West is separated from Kakula by the West Scarp Fault and is planned as an independent mine. The Kakula West and the Kamoa North Mines 1–5, PEA analyses have been prepared using the Mineral Resources stated in the Kamoa-Kakula 2020 Resource Update.

The potential development scenarios at the Kamoa-Kakula Project include the Kamoa-Kakula IDP20 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 3.8 Mtpa is expanded to 7.6 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 11.4 Mtpa combined by Year-6. The next phase of development described by the Kamoa-Kakula 2020 PEA is from Kakula West followed by five new mines at Kamoa North to bring total production to 19.0 Mtpa.

Construction is well underway on the Kakula mine and the first 3.8 Mtpa module of the mill with long-lead time orders placed for the second module, necessitating mine development at Kansoko. The immediate decision for Ivanhoe and its partners is to Kamoa Copper will make further decisions to determine the sequence for ramping up the production rate. 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 IDP20 production and economic analysis results are shown in Table 1.1.

Figure 1.2 Kamoa-Kakula IDP20 Long-Term Development Scenario

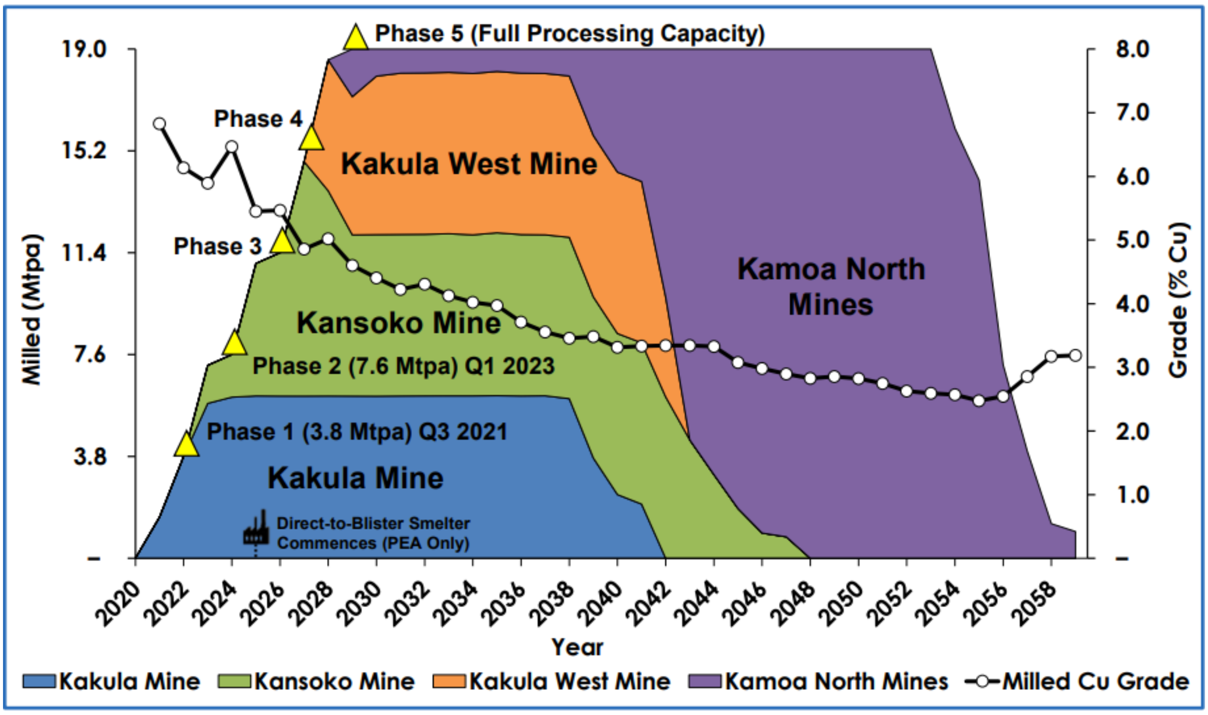


Figure by OreWin, 2020.

Figure 1.3 Kamoa-Kakula IDP20 Site Plan

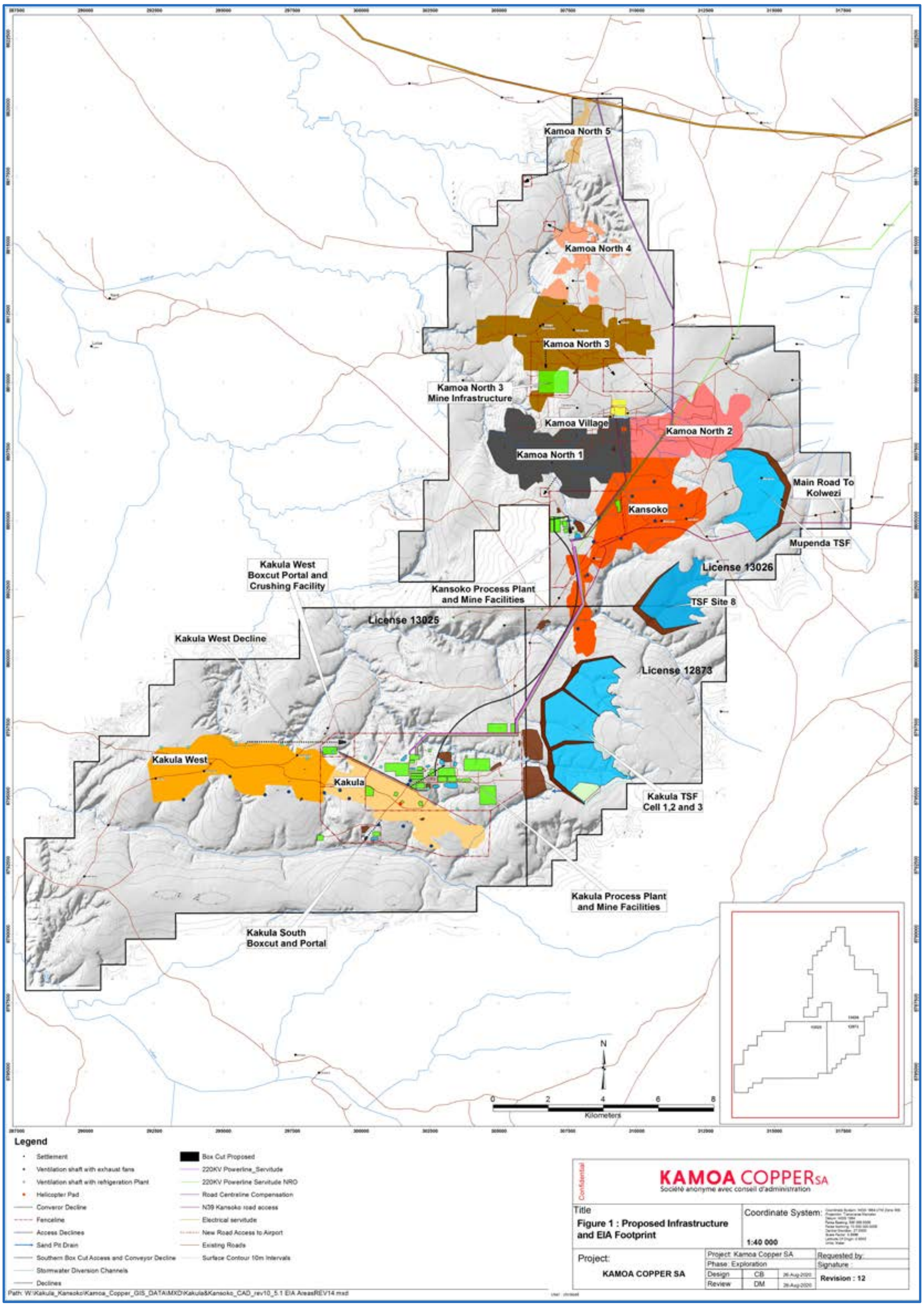


Figure by Kamoa Copper SA, 2020.

Table 1.1 Kamoa-Kakula IDP20 Results Summary

Item	Unit	Kakula 2020 FS	Kakula-Kansoko 2020 PFS	Kamoa-Kakula 2020 PEA
Total Processed				
Quantity Milled	kt	109,975	235,157	597,621
Copper Feed Grade	%	5.22	4.47	3.63
Total Concentrate Produced				
Copper Concentrate Produced	kt (dry)	8,542	19,948	42,818
Copper Concentrate - External Smelter	kt (dry)	8,542	19,948	11,944
Copper Concentrate - Internal Smelter	kt (dry)	–	–	30,874
Copper Recovery	%	85.23	86.27	86.42
Copper Concentrate Grade	%	57.32	45.49	43.76
Contained Copper in Conc. - External Smelter	Mlb	10,795	20,006	13,251
Contained Copper in Conc. - External Smelter	kt	4897	9,075	6,010
Contained Copper in Blister - Internal Smelter	Mlb	–	–	27,641
Contained Copper in Blister - Internal Smelter	kt	–	–	12,538
Peak Annual Recovered Copper Production	kt	366	427	805
10-Year Average				
Copper Concentrate Produced	kt (dry)	496	622	1,043
Contained Copper in Conc. - External Smelter	kt	284	331	248
Contained Copper in Blister - Internal Smelter	kt	–	–	253
Mine-Site Cash Cost (Including Smelter)	US$/lb Cu	0.52	0.55	0.65
Total Cash Cost	US$/lb Cu	1.16	1.23	1.07
Key Financial Results				
Peak Funding	US$M	775	848	784
Initial Capital Costs	US$M	646	695	715
Expansion Capital Costs	US$M	594	750	4,461
Sustaining Capital Cost	US$M	1,265	2,827	11,958
Mine Site Cash Cost	US$/lb Cu	0.62	0.64	0.92
Total Cash Costs After Credits	US$/lb Cu	1.26	1.44	1.28
Site Operating Costs	US$/t Milled	58.73	52.95	62.44
After-Tax NPV8%	US$M	5,520	6,604	11,117
After-Tax IRR	%	77.0	69.0	56.2
Project Payback	Years	2.3	2.5	3.6
Project Life	Years	21	37	43

1.3 Project Development

The initial plant capacity requires a ramp-up of mining operations to create pre-production stockpiles that will be used for the processing plant to ramp-up to a steady state throughput of 3.8 Mtpa.

Underground development is currently underway at Kakula via twin declines in the north and a single decline in the south. Approximately 23 km of development had been achieved by the end of September 2020. As of the end of September 2020, pre-production stockpiles held approximately 27,000 t of contained copper, which is approximately 13% ahead of schedule based on the Kakula 2020 FS. Underground infrastructure construction is in progress with the first ventilation shaft complete, and three others shafts in various stages of completion. The main decline dams and pump station have been completed. The decline rock handling system including tips, bins, feeders and the main decline conveyor was also recently completed, which. Once operational, the rock handling system will allow stope development to commence once a main access drive connecting the north and south declines is completed, targeted for November 2020. Surface infrastructure to support mining activities at the north and south portals is functional with construction of permanent facilities underway.

Basic engineering design for the first phase of project development was completed in July 2019. Following the completion of basic engineering, the initial processing plant capacity increased from 3.0–3.8 Mtpa. Phase 1 of the Project consists of a 3.8 Mtpa mine and concentrator. The concentrator has been designed using a modular approach with the second 3.8 Mtpa stream to be constructed as the mine ramps up, and subject to the availability of funding. Work has commenced on the second stream, and equipment with long lead items have been ordered.

Orders for all processing plant long lead items were placed in July 2019. These items include; ball mills, flotation cells, crushers, HPGR (high-pressure grinding rolls), concentrate filter, thickeners, and regrind mills. All long lead items have arrived on site as of the end of September 2020, with the exception of the transformers. The plant earthworks and terracing were completed at the end of 2019 and earthworks for surface infrastructure is well advanced. Civil work for the processing plant is underway nearing completion, including with the main focus areas being the stockpiles and mill foundations. The contract for the supply, fabrication and erection of steel, platework, piping, and mechanical equipment has been awarded and majority of the steel, platework and piping is on site and steel erection is well advanced.

Several surface infrastructure facilities were completed by the end of 2019. These include the main access road linking the mine to Kolwezi airport and the 1,000 bed Kakula village. Work on the construction of the main power supply to the mine consisting of a 220 kV powerline and substation is underway. The on-mine electrical infrastructure construction including sub stations, transformers, and power lines have been designed and construction commenced.

The updated estimate of the project’s initial capital costs is approximately US$1.3 billion (estimate 1 January 2019). This assumes commissioning of the processing plant in Q3’21. The capital cost incurred by the Project during 2019 was US$309M and in the first half of 2020 was US$243M.

1.4 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 a recently completed 42 km unsealed road via Kolwezi airport, which by-passes major settlements. 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–October) and wet (November–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 several years.

1.5 Project Ownership

The Project titles consists of three exploitation licences (Exploitation Permit No. 12873, 13025, and 13026) which cover an area of 397.4 km^2) title of the exploitation licences is held by Kamoa Copper. The Exploitation Licences were approved 20 August 2012, and grant Kamoa Copper the right to explore for, develop and exploit copper and other minerals for an initial 30-year term. The licences expire 19 August 2042 but can then be extended for 15-year periods, until the end of the mine's life.

Kamoa Copper is a subsidiary of Kamoa Holding Limited (80%) and the DRC Government (20%).

Kamoa Holding Limited is owned by Ivanhoe (49.5%), Gold Mountains (H.K.) International Mining Company Limited (49.5%), and Crystal River Global Limited (1%).

Gold Mountains (H.K.) International Mining Company Limited (49.5%) is a subsidiary of Zijin Mining Group Co., Ltd (Zijin).

Crystal River Global Limited is a private company.

The relationship between Ivanhoe, Zijin, and Crystal River Global Limited is governed by a shareholder, governance and option agreement which provides for Kamoa Holding Limited's Board of Directors to make all key decisions regarding the development and operation of the Project.

1.5.1 Ownership History

In September 2012, a 5%, non-dilutable interest in Kamoa Copper was transferred to the government of the DRC. This transfer was pursuant to the DRC mining code and for no consideration.

In December 2015, Zijin acquired a 49.5% share interest in Kamoa Holding, through its subsidiary Gold Mountains International Mining Company Limited. This share was acquired from Ivanhoe for an aggregate cash consideration of US$412M.

In November 2016, Ivanhoe, Zijin and the DRC Government signed an agreement that transferred an additional 15% interest in the Kamoa Copper to the DRC Government. This increased the DRC Government total stake in the Project to 20%.

The current effective ownership of Kamoa Copper and the Project is Ivanhoe (39.6%), Zijin (39.6%) and Crystal River Global Limited (0.8%), which each hold an indirect interest through ownership of Kamoa Holding Limited, and the DRC Government which holds a direct 20% interest in Kamoa Copper.

1.6 Mineral and Surface Rights, Royalties, and Agreements

Land access for exploration programmes and project development completed to date has been negotiated without problems. Where compensation has been required for exploration activities, compensation has followed International Finance Corporation (IFC) and World Bank guidelines.

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 consider 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.

1.7 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 Conglomérat, which is stratigraphically higher than the majority of Copperbelt deposits, which are typically hosted by dolomitic rocks of the Mines Subgroup.

The metallogenic province of the Central African Copperbelt is hosted in metasedimentary rocks of the Neoproterozoic Katanga Basin, an evolving intracontinental rift. The Katangan Basin overlies a composite basement over which the lowermost, continental siliciclastic rock sequences within the Katangan Basin were deposited in a series of restricted rift basins that were then overlain by laterally extensive, organic-rich, marine siltstones and shales. The metasedimentary rocks that host the Central African Copperbelt mineralisation form a sequence known as the Katanga Supergroup, comprising the Roan, N'Guba, and 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 N'Guba 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 that host the 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 diamictite (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. At Kamoa, mineralisation thicknesses at a 1.0% Cu cut-off grade range from 2.3–21.6 m (for Indicated Mineral Resources). At Kamoa North, a locally developed zone of high-grade copper mineralisation, known as the Bonanza Zone, dips at approximately 40º, parallel to the Bonanza Fault, and is hosted within the Kamoa Pyritic Siltstone (KPS). At a 1.0% Cu cut-off, it ranges in true thickness from <1–24.0 m (for Indicated Mineral Resources) and remains open to the west. Hypogene mineralisation is characterised by chalcopyrite and bornite-dominant zones. There is significant pyrite mineralisation in the KPS above the mineralised horizon that could possibly be exploited to produce pyrite concentrates for sulfuric acid production.

At the Kakula deposit, a deeper basinal setting has resulted in significant thickening of the diamictite basal units with the development of several 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 sulfides in the mineralised sequence are zoned with chalcocite (Cu_2S), bornite (Cu_5FeS_4) and chalcopyrite ($CuFeS_2$), with chalcocite being the dominant mineral. At Kakula, mineralisation thicknesses at a 1.0% Cu cut-off grade range from 2.9–42.5 m (for Indicated Mineral Resources).

Copper mineralisation comprises three distinct styles: supergene, hypogene, and mixed. Near the surface adjacent to the domes, the diamictites have been leached, resulting in zones of copper oxides and secondary copper sulfide 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 the mineralisation occurs well below the surface and is hypogene.

1.8 Exploration

Exploration was undertaken in the current Project area by the Tenke Fungurume Consortium between 1971–1975. Although a localised regional stream-sediment sampling may have been performed, no information is available from this study.

Work performed from 2003 to date by Ivanhoe and its third-party contractors on the Project has included geological mapping, geochemical sampling, airborne geophysical surveys, ground geophysical surveys, reverse circulation (RC), and core drilling, and petrographic studies.

Exploration activities at the Kamoa-Kakula Project have been 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), and four 2D seismic lines have been completed. The seismic survey was designed to locate the top of the Roan, interpret broad-scale basin architecture and locate the growth faults and younger brittle structures. Several other geophysical studies such as ground gravity, ground magnetics and “Excalibur” airborne surveys were conducted in the Kamoa North area in 2019 to better understand the controls of the higher-grade mineralisation.

Integration of the geophysical programme results with the geological models supported detailed exploration targeting within the highly prospective Kamoa-Kakula exploitation licence area. These have also resulted in the Bonanza Zone and Far North discoveries being incorporated into the Kamoa resource model. Several geophysical studies such as ground gravity, ground magnetics and “Excalibur” airborne were conducted in the Kamoa North area in 2019 to better understand the controls of the ultra-high-grade mineralisation to assist in locating additional targets.

In the opinion of the Wood Qualified Person (the Wood QP), 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 that additional work is warranted in the Project area.

1.9 Drilling

The drillhole database used for the Kamoa resource estimate was closed on 20 January 2020. The resource model for Kamoa was updated as of 30 January 2020. The drillhole database used for the Kakula resource estimate was closed on 1 November 2018, and the model was completed as of 10 November 2018.

Core holes 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 18 September 2020, there were 2,159 core holes drilled within the Kamoa-Kakula Project. The Kamoa Mineral Resource estimate of January 2020 is supported by 998 drillholes. Included in the 998 drillholes were 17 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 354 drillhole intercepts (one intercept per drillhole).

The 807 holes not included in either the January 2020 Kamoa or the November 2018 Kakula estimate were excluded because they were abandoned, unmineralised holes in the dome areas, unsampled underground cover, 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 (20 January 2020), 79 drillholes have been completed inside of the modelled area at Kamoa. An additional 85 drillholes at Kakula have been completed after the closure of the database on 1 November 2018, primarily for geotechnical purposes, or as infill drilling in close proximity to the current underground development.

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 0–100% and averages 95%. Visual inspection of the Kamoa core by the Wood QP 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. As of 20 January 2020, only three drillholes used in the Mineral Resource estimate (DKMC_DD1580, DKMC_DD1600 and DKMC_DD1621) lacked an independently surveyed collar position.

The Kakula drillhole collars have been surveyed by SD Geomatique and E.M.K. Construction SARL. As of 1 November 2018, there were no outstanding collar surveys being used in the Kakula Mineral Resource estimate. Visual inspection of the Kakula core by the Wood QP documented the core recovery to be excellent.

In the opinion of the Wood QP, 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.

1.10 Sample Preparation, Analyses, and Security

Whole core is logged by the geologist on major lithological intervals, until mineralised material or a "zone of interest" (ZI) such as a lithology that is conventionally sampled (e.g. the Kamoa Pyritic Siltstone) is encountered. The ZI is logged on sampling intervals, typically 1 m intervals (dependent on geological controls). Within any ZI, 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, including 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 2010–2018 drilling. ALS Limited is ISO:9001:2008 registered and ISO:17025-accredited. Check sample selection for the 2019 and 2020 drilling is currently being prepared.

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 (500–1,000 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. Acid-soluble copper (ASCu) assays have been primarily undertaken at Kamoa since 2010.

Ivanhoe has discontinued ASCu analysis at Kakula, with no ASCu analysis for the vast majority of Kakula drillholes. The discontinuation results from all the mineralisation at Kakula being considered to be hypogene.

In the opinion of the Wood QP, 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.

1.11 Data Verification

Wood reviewed the sample chain of custody, quality assurance and control (QA/QC) procedures, and qualifications of analytical laboratories. Wood is of the opinion that the procedures and QA/QC control are acceptable to support Mineral Resource estimation. Wood also audited the assay database, core logging, and geological interpretations on a number of occasions between 2009–2020, including during site visits conducted by the Wood QP, and has found no material issues with the data as a result of these audits. Independent witness sampling and assaying programs conducted by Wood were found to be consistent with Ivanhoe's original sampling and assaying.

In the opinion of the Wood QP, 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.

1.12 Metallurgical Testwork

The Kamoa-Kakula resource has a long history of metallurgical testwork (2010–2015) 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).

The initial mineralogical and flotation testwork on the Kakula resource was conducted during 2016 and 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 3.96–8.19%.

Following the successful preliminary testing of the Kakula samples, additional drill core material was tested in 2017 and 2018 for a prefeasibility study testwork campaign, which focussed on flow sheet optimisation. Testwork completed during 2017 and 2018 included: various mineralogical studies, comminution parameter testing, flotation flow sheet optimisation, high pressure grinding rolls (HPGR) testwork, concentrate and tailings thickening, filtration testing, bulk material flow testwork, comminution variability testwork, and flotation variability testwork.

XPS was contracted by Kamoa Copper SA in March 2019 to conduct a mini pilot plant (MPP) campaign, to generate sample for various testwork campaigns, utilising reject comminution sample at a grade of 7.1% Cu. The flow sheets used during the sample preparation was modified throughout the campaign to prioritise the production of the various samples, as required at the time.

The recovery estimate for the Kakula 2020 FS is based on the test information generated by the final duplicate testing of the Kakula 2019 PFS flow sheet, as well as the Kakula variability campaign. The recovery model is based on a final product grade of 57.32% Cu. The average Cu recovery over the life of mine to produce a 57.32% Cu concentrate was determined as 85.23% from a 5.22% Cu head grade.

1.13 Mineral Resource Estimates

The Kamoa and Kakula resource models are based on the same 3D estimation methodology. The location of the resource models are shown in relation to the context of the broader Project in Figure 1.4.

The Kamoa resource model was controlled using 11 domains based on a combination of stratigraphy and a mineralised envelope. The Kakula resource model was controlled using six domains based on a combination of stratigraphy and a mineralised envelope. For both Kamoa and Kakula, the mineralised envelope was defined using an approximate cut-off grade of 1% TCu.

To account for the undulations of the deposits and ensure that the vertical 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 mineralisation. This method aligns the top, middle and bottom of the mineralised intervals horizontally for variography and grade estimation using ordinary kriging (OK). This preserves the important vertical grade profile 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.

At a 1% copper cut-off, the Kamoa Mineral Resources cover an area of 55.2 km^2 (Indicated Mineral Resource) and 21.8 km^2 classified as Inferred Mineral Resources. The deposit remains open laterally.

At a 1% copper cut-off, the Kakula Mineral Resources cover an area of 21.7 km^2 (Indicated Mineral Resource) and 5.6 km^2 classified as Inferred Mineral Resources. The deposit remains open laterally, and the southern parts of the Kamoa-Kakula exploitation licence area is virtually untested.

For reporting Mineral Resources, Wood 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 selective mineralised zone (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, Wood performed a conceptual analysis.

Figure 1.4 Kamoa-Kakula Exploitation Licence, Showing the Kamoa, Kakula, Kakula West, Kamoa North Bonanza Zone and Kamoa Far North Mineral Resource Areas.

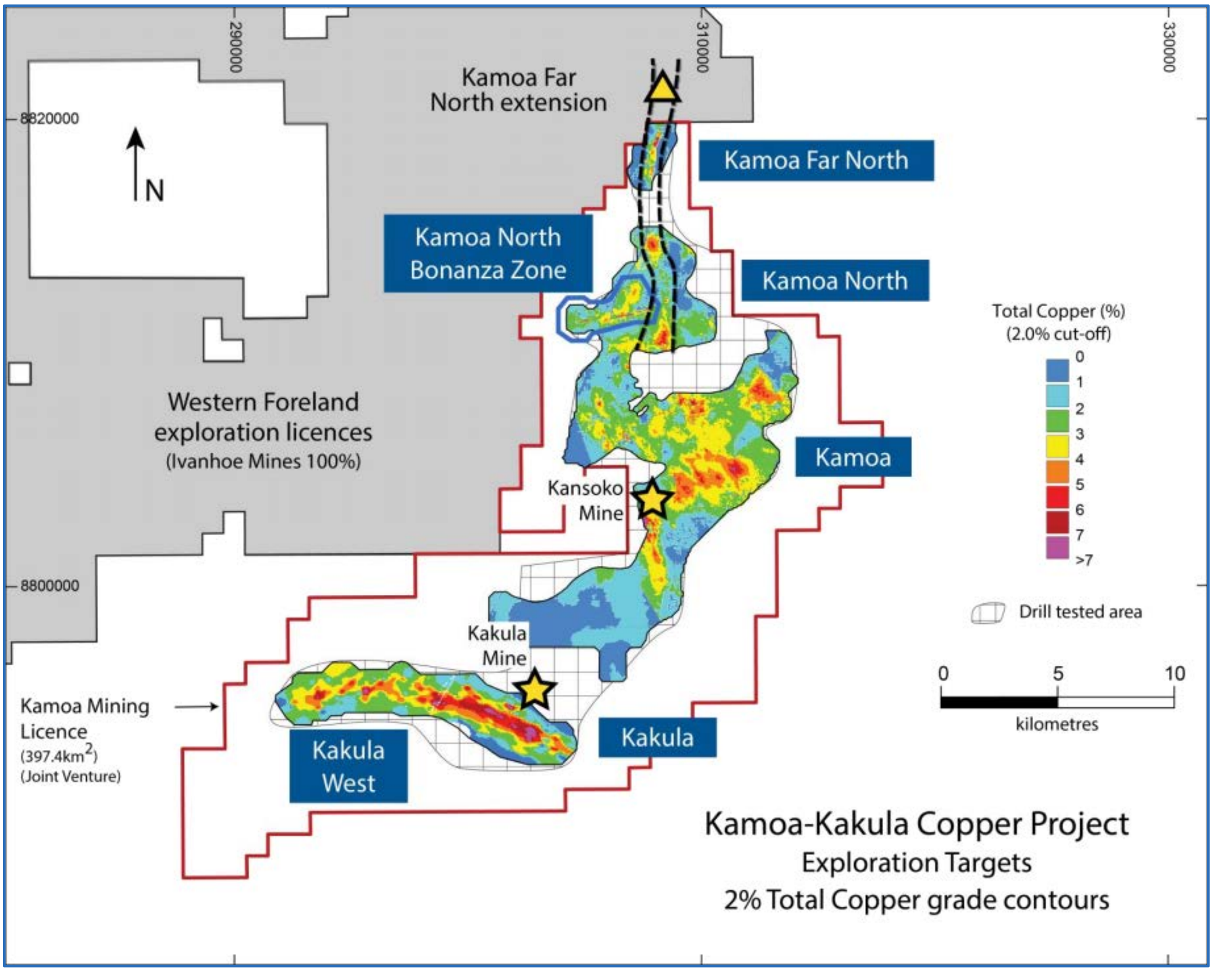


Figure by Ivanhoe, 2020.

1.13.1 Kamoa-Kakula Mineral Resource Statement

The effective date of the estimate for Kamoa is 30 January 2020 and the cut-off date for drill data is 20 January 2020. The Kakula Mineral Resources were estimated as of 10 November 2018. On 10 February 2020, the inputs used in assessing reasonable prospects of eventual extraction and the drill data inputs were reviewed to ensure the estimate remained current. There are no changes to the estimate as a result of the review, and the estimate has an effective date of 10 February 2020.

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 Kamoa Mineral Resource at a range of cut-off grades with the base case cut-off of 1.0% TCu highlighted in grey. Table 1.4 summarises the Kakula Mineral Resource at a range of cut-off grades with the base case cut-off of 1.0% TCu highlighted. Mineral Resources in Table 1.3 and Table 1.4 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 (km^2)	Copper (%)	Vertical Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
Kamoa	Indicated	760	55.2	2.73	5.0	20,800	45.8
	Inferred	235	21.8	1.70	4.0	4,010	8.8
Kakula	Indicated	627	21.7	2.74	10.3	17,200	37.9
	Inferred	104	5.6	1.61	6.7	1,680	3.7
Total Kamoa-Kakula Project	Indicated	1,387	77.0	2.74	6.5	38,000	83.7
	Inferred	339	27.4	1.68	4.5	5,690	12.5

1. Ivanhoe's Vice President Resource, 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 Gordon Seibel, a Registered Member (RM) of the Society for Mining, Metallurgy and Exploration (SME), who is the Qualified Person for the Mineral Resource estimate. The effective date of the estimate for Kamoa is 30 January 2020, and the cut-off date for drill data is 20 January 2020. The Mineral Resources at Kakula were estimated as of 10 November 2018, and the cut-off date for the drill data is 1 November 2018. On 10 February 2020, the inputs used in assessing reasonable prospects of eventual extraction and the drill data inputs were reviewed to ensure the estimate remained current. There are no changes to the estimate as a result of the review, and the estimate has an effective date of 10 February 2020. Mineral Resources are reported using the CIM 2014 Definition Standards for Mineral Resources and Mineral Reserves. Mineral Resources 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.
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 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, 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 hanging wall 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 Kamoa: Sensitivity of Mineral Resources to Cut-off Grade

Cut-off (% Cu)	Tonnage (Mt)	Area (km²)	Copper (%)	True Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
Indicated Mineral Resource						
5.0	44	4.5	6.14	3.5	2,690	5.9
4.5	67	6.7	5.65	3.6	3,800	8.4
4.0	107	10.4	5.13	3.7	5,490	12.1
3.5	171	16.4	4.61	3.7	7,890	17.4
3.0	256	24.0	4.15	3.8	10,700	23.5
2.5	369	32.8	3.73	4.1	13,700	30.3
2.0	504	41.5	3.33	4.4	16,800	37.0
1.5	655	49.4	2.97	4.8	19,400	42.8
1.0	760	55.2	2.73	5.0	20,800	45.8
0.5	1,185	59.4	1.99	7.3	23,600	52.0
Inferred Mineral Resource						
4.0	1	0.1	5.47	3.4	55	0.1
3.5	4	0.5	4.12	3.1	177	0.4
3.0	13	1.5	3.51	3.1	441	1.0
2.5	30	3.5	3.08	3.0	910	2.0
2.0	58	6.5	2.66	3.2	1,540	3.4
1.5	113	11.9	2.20	3.4	2,480	5.5
1.0	235	21.8	1.70	4.0	4,010	8.8
0.5	680	31.4	1.01	8.0	6,860	15.1

1. Ivanhoe's Vice President, Resources 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 Gordon Seibel, a Registered Member (RM) of the Society for Mining, Metallurgy and Exploration (SME), employee of Wood, who is the Qualified Person for the Mineral Resource estimate. The effective date of the estimate is 30 January 2020 and the cut-off date for drill data is 20 January 2020. Mineral Resources are reported using the CIM 2014 Definition Standards for Mineral Resources and Mineral Reserves. Mineral Resources 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.

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 hanging wall or footwall contact boundary loss and dilution. No mining recovery has been applied.
4. Depth of mineralisation below the surface ranges from 10–1,320 m for Indicated Mineral Resources and 20–1,560 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.
7. This table is not additive to Table 1.2

Table 1.4 Kakula: Sensitivity of Mineral Resources to Cut-off Grade

Cut-off (% Cu)	Tonnage (Mt)	Area (km^2)	Copper (%)	True Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
Indicated Mineral Resource						
5.0	77	5.9	7.48	4.5	5,730	12.6
4.5	91	7.0	7.04	4.5	6,440	14.2
4.0	109	8.3	6.58	4.6	7,200	15.9
3.5	132	9.9	6.09	4.7	8,060	17.8
3.0	167	11.8	5.50	5.0	9,180	20.2
2.5	218	14.3	4.85	5.4	10,600	23.3
2.0	318	17.5	4.02	6.5	12,800	28.2
1.5	435	19.6	3.41	7.9	14,900	32.7
1.0	627	21.7	2.74	10.3	17,200	37.9
0.5	939	22.6	2.08	14.9	19,500	43.0
Inferred Mineral Resource						
4.0	1	0.1	4.41	3.3	33	0.1
3.5	2	0.2	4.04	3.6	67	0.1
3.0	5	0.4	3.52	3.9	168	0.4
2.5	10	1.0	3.10	3.7	324	0.7
2.0	22	2.0	2.64	3.9	583	1.3
1.5	45	3.7	2.18	4.3	974	2.1
1.0	104	5.6	1.61	6.7	1,680	3.7
0.5	257	7.9	1.08	11.7	2,770	6.1

1. Ivanhoe's Vice President, Resources 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 Gordon Seibel, a Registered Member (RM) of the Society for Mining, Metallurgy and Exploration (SME), employee of Wood, who is the Qualified Person for the Mineral Resources. The Mineral Resources at Kakula were estimated as of 10 November 2018 and the cut-off date for the drill data is 1 November 2018. On 10 February 2020, the inputs used in assessing reasonable prospects of eventual extraction and the drill data inputs were reviewed to ensure the estimate remained current. There are no changes to the estimate as a result of the review, and the estimate has an effective date of 10 February 2020. Mineral Resources are reported using the CIM 2014 Definition Standards for Mineral Resources and Mineral Reserves. Mineral Resources 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.
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 hanging wall or footwall contact boundary loss and dilution. No mining recovery has been applied.
4. Depth of mineralisation below the surface ranges from 12–1,373 m for Indicated Mineral Resources and 61–1,397 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 differences between tonnes, grade and contained metal content.
7. This table is not additive to Table 1.2.

1.13.2 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 underground development 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.
  - At the Kakula deposit, the mineralisation appears more continuous compared to Kamoa.
  - In the Kakula south developments, minor offsets across growth faults have been encountered, but adjustments to the he mining methods has allowed the mining to follow the steeper dips of the mineralisation across the faults.
  - In the Kakula northern access drive, a larger growth fault was encountered where the mineralisation of the south side of the fault was faulted down (with variable offsets). A spiral decline was developed to accommodate the offsets, and re-established mining on the mineralisation.
  - The Kakula southern and northern declines and associated development are expected to join towards the end of 2020. This will provide a complete section across the deposit to further study the structural geology of the Kakula deposit.
- 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.

- 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.

1.14 Targets for Further Exploration

Specific targets for further exploration are not currently defined at Kamoa-Kakula. The eastern boundary of the Mineral Resources at Kamoa is defined solely by the current limit of drilling, at depths ranging from 600–1,560 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.

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. There is excellent potential for discovery of additional mineralisation.

1.15 Mineral Reserves

1.15.1 Kamoa-Kakula Project Mineral Reserve

The Kamoa-Kakula Project Mineral Reserve includes 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 Reserve

Classification	Ore (Mt)	Copper (%)	Contained Copper (Mlb)	Contained Copper (kt)
Proven Kakula Mineral Reserve	–	–	–	–
Probable Kakula Mineral Reserve	110	5.22	12,665	5,745
Proven Kansoko Mineral Reserve	–	–	–	–
Probable Kansoko Mineral Reserve	125	3.81	10,525	4,774
Total Proven Mineral Reserve	–	–	–	–
Total Probable Mineral Reserve	235	4.47	23,190	10,519

1. Effective date of the all Mineral Reserves is 8 September 2020.
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$3.10 for Kakula and US$3.00/lb for Kansoko.
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.

1.15.2 Kakula 2020 Mineral Reserve

The Kakula 2020 Mineral Reserve is defined by a feasibility level study. The Mineral Reserve targeted in the mine plan focused on maximising the grade profile for a 6.0 Mtpa full production rate for 15-years, a 4-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 110 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 2020 FS Probable Mineral Reserves are shown in Table 1.6.

Table 1.6 Kakula 2020 FS Mineral Reserve

Classification	Ore (Mt)	Copper (%)	Contained Copper (Mlb)	Contained Copper (kt)
Proven Mineral Reserve	–	–	–	–
Probable Mineral Reserve	110	5.22	12,665	5,745
Mineral Reserve	110	5.22	12,665	5,745

1. Effective date of the Kakula Mineral Reserves is 8 September 2020.
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$3.10/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.

1.15.3 Kansoko 2020 Mineral Reserve

The Kansoko Mineral Reserve estimate is defined by a prefeasibility study.

The reserve focused on utilising the 7.6 Mtpa processing capacity. As such, a range of NSR cut-offs were evaluated to develop the reserve statement to get approximately 125 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 life-of-mine (LOM) schedule resulted in 18-years with a production rate of 1.6 Mtpa and 15-years at a full production of 6.0 Mtpa.

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 2020 PFS are summarised in Table 1.7.

Table 1.7 Kamoa 2020 PFS Mineral Reserve

Classification	Tonnage (Mt)	Copper (%)	Contained Copper (Mlb)	Contained Copper (kt)
Proven Mineral Reserve	–	–	–	–
Probable Mineral Reserve	125	3.81	10,525	4,774
Mineral Reserve	125	3.81	10,525	4,774

1. Effective date of the Kamoa Mineral Reserve is 8 September 2020.
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.

1.16 Kakula 2020 FS

Ivanhoe has developed 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 2020 FS is the construction and operation of an underground mine, concentrator processing facilities, and associated infrastructure. The Kakula 2020 FS production is planned to be 6.0 Mtpa ore with a mine life of 21-years.

1.16.1 Kakula 2020 FS Mining

The Kakula 2020 FS 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 dimensions of 7 m W x 6 m H, and the service decline has dimensions of 5.5 m W x 6 m H. The south decline will serve as a secondary operational ingress/egress and will facilitate critical early mine development. The south decline has dimensions of 7.5 m W x 6 m H. 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 1.5. The mining methods for the Kakula deposit are drift-and-fill using paste backfill and room-and-pillar. The paste backfill system will utilize a paste plant located on surface connected to a distribution system that includes a surface pipe network connected to bore holes located at each connection drive on the North side of the orebody. Approximately 99% of the deposit will be mined using drift-and-fill; with the exception of a room-and-pillar area close to the north declines, which will be mined in the early years of production. Table 1.8 shows the Kakula 2020 FS Mineral Reserves by mining method.

Table 1.8 Kakula 2020 FS Mineral Reserves by Mining Method

Production by Mining Method	Tonnes (Mt)	Metres (km)	NSR ($/t)	Cu (%)
Ore Development	11.0	105	243.12	5.22
Drift and Fill	80.4	938	240.31	5.16
Pillar Extraction	17.2	197	263.11	5.64
Room and Pillar	1.4	17	161.61	3.52
Total	110.0	1,257	243.18	5.22

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

Figure 1.5 Kakula 2020 FS Mine Development

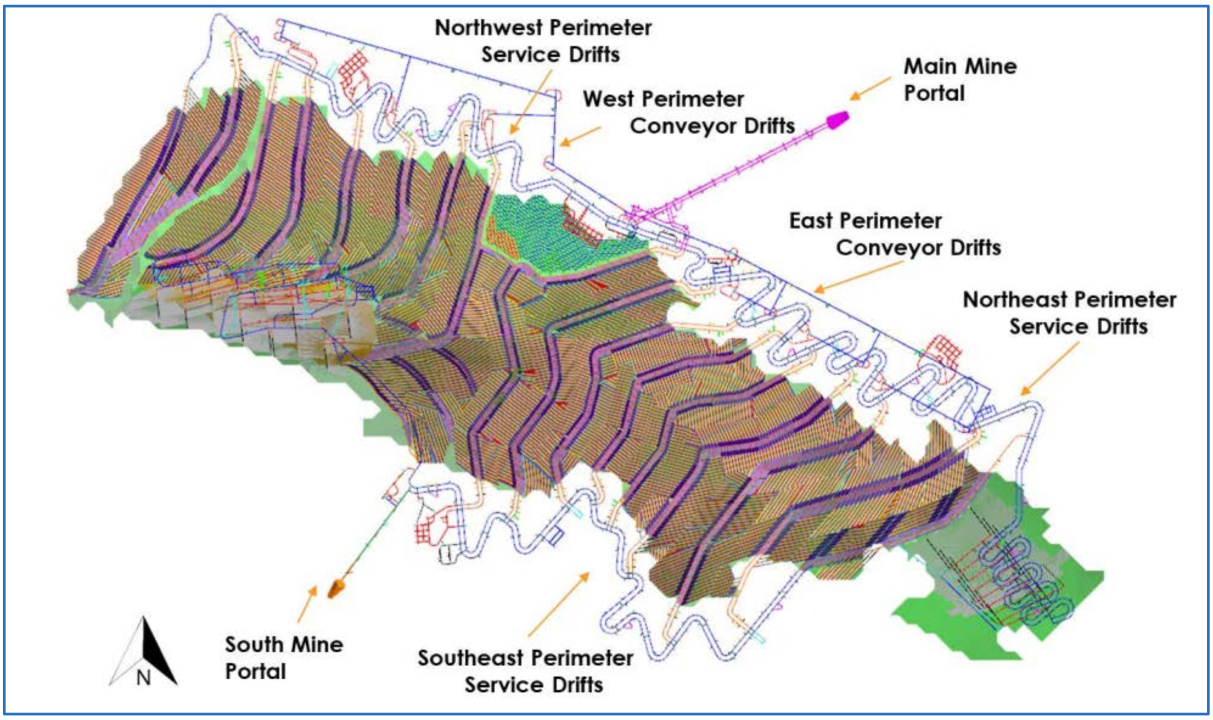


Figure by Stantec, 2020.

1.16.2 Kakula 2020 FS Process

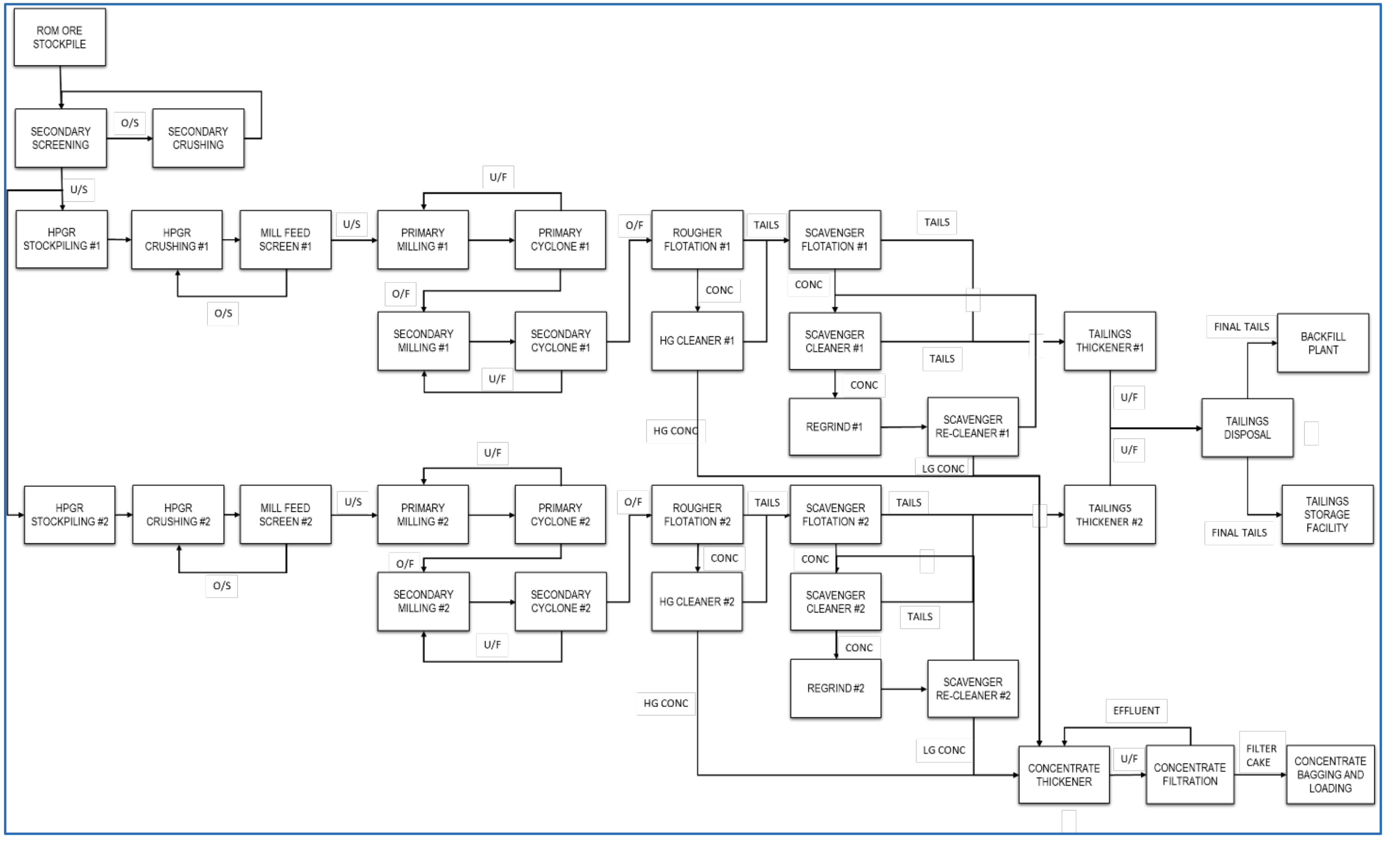
The Kakula concentrator are constructed in a phased approach with two 3.8 Mtpa modules to finally have a capacity of 7.6 Mtpa.

The flow sheet for the Kakula concentrator is shown in Figure 1.6. The Kakula concentrator design incorporates a run-of-mine stockpile, followed by cone crushers operating in closed circuit with vibrating screens to produce 100% passing 50 mm material that is stockpiled. The crushed ore is fed to the High Pressure Grinding Rolls (HPGR) operating in closed circuit with wet screening, at a product size of 80% (P_{80}) passing 4.5 mm which is gravity fed to the milling circuit. The milling circuit incorporates two stages of ball milling in series in closed circuit with cyclone clusters for further size reduction and classification to a target grind size of 80% passing 53 micrometres (µm). The milled slurry is pumped to the rougher and scavenger flotation circuit where the high-grade, or fast-floating rougher concentrate, and medium-grade, or slow-floating scavenger concentrate, are separated for further upgrading. The rougher concentrate is upgraded in the low entrainment high-grade cleaner stage to produce a high-grade concentrate. The medium-grade or scavenger concentrate together with the tailings from the high-grade cleaner stage and the recycled scavenger recleaner tailings are combined and further upgraded in the scavenger cleaner circuit.

The concentrate produced from the scavenger cleaner circuit, representing roughly 15% of the mill feed, is re-ground to a P_{80} of 10 µm prior to final cleaning in the low entrainment scavenger recleaner stage. The scavenger recleaner concentrate is then combined with the high-grade cleaner concentrate to form final concentrate. The final concentrate is then thickened and pumped to the concentrate filter. Final filtered concentrate is then bagged for shipment to market. The scavenger tailings and scavenger cleaner tailings are combined and thickened prior to being pumped to the backfill plant and/or to the tailings storage facility. Backfill utilizes on average 40% of tailings and the remaining 60% will be pumped to the tailings storage facility.

Based on extensive testwork, the concentrator is expected to achieve an overall recovery of 85%, producing a high-grade concentrate grading 57% copper. Kakula also benefits from having very low deleterious elements, including arsenic levels of 0.02%.

Figure 1.6 Kakula Process Block Flow Diagram



1.16.3 Kakula 2020 FS Transport

During the initial production years, the North South corridor between southern DRC and Durban or Richards Bay in South Africa is considered the most attractive and reliable export corridor. Bagged concentrate product will be packed on site and transported by truck to Ndola in Zambia, where it will be loaded onto trains and transported to the Durban or Richards Bay port. Alternatively, it is possible to transport concentrate to the ports of Dar es Salaam in Tanzania, or Walvis Bay in Namibia.

In addition an export road (Katonoto export road) is under construction that will allow export trucking to be done via a main export road connecting to the west to the Angolan rail / road network and the ports of Lobito or Luanda.

On-site infrastructure is planned to support the import / export logistics including a large secured truck park area, bonded yard for customs clearing, weigh bridges, and product sampling systems.

A new 30 km access road has been constructed by Kamoa from Kolwezi to the Kakula Mine. This road has been designed for high traffic volumes and for relatively high speeds. It is a gravel road, but the foundations are suitable for later asphalting of the road. This road will provide a direct link to the main road to Lubumbashi and other industries in the region including a copper smelter and cement plants. This road will also reduce commuting times between Kolwezi and Kakula to 30 minutes, making it feasible for much of Kakula's labour to commute daily from Kolwezi.

1.16.4 Kakula 2020 FS Results

The Kakula 2020 FS describes the initial phase of the Kakula development and presents the first Mineral Reserve for the Kakula Deposit. The Kakula 2020 FS evaluates the development of a 6.0 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 1.7.

The Kakula 2020 FS has an average annual production rate of 284,000 t of copper at a mine site cash cost of US$0.52/lb copper and total cash cost of US$1.16/lb copper for the first ten years of operations. The preproduction capital cost of US$646M for this option would result in an after-tax net present value at an 8% discount rate (NPV8%) of US$5.52 billion.

Figure 1.7 Kakula 2020 FS 6.0 Mtpa Development Scenario

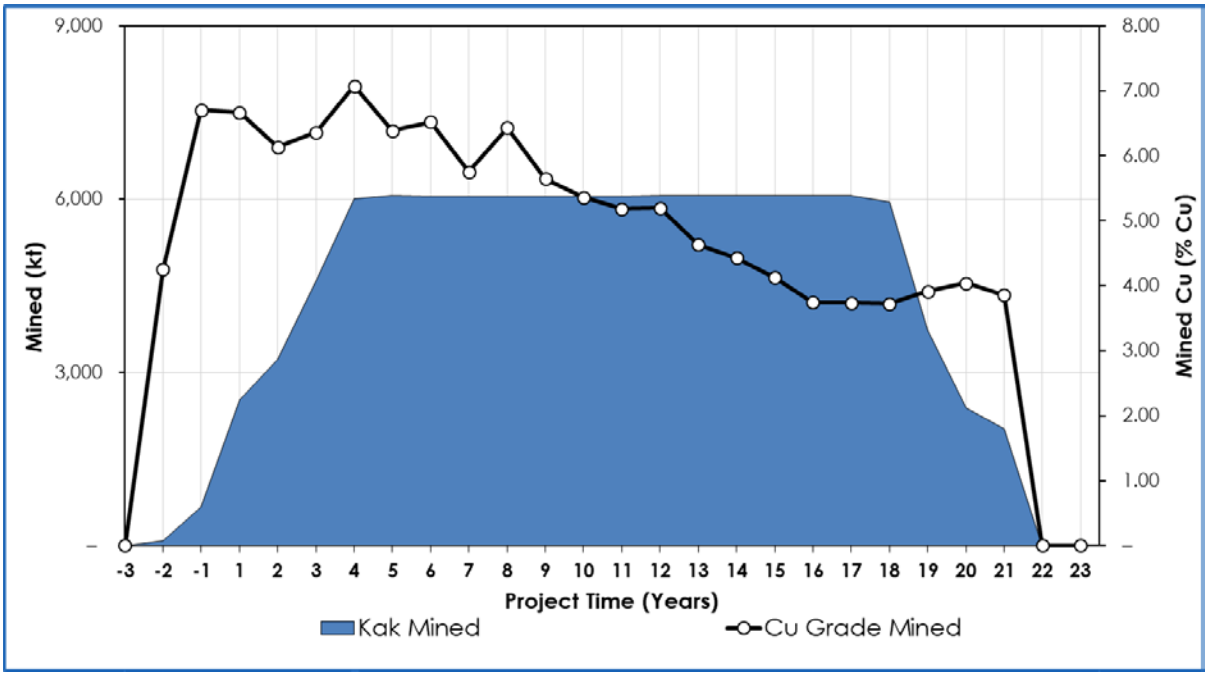


Figure by OreWin, 2020.

A summary of the key results for the Kakula 2020 FS scenario are:

- Very-high-grade initial phase of production is projected to have a grade of 7.1% copper in Year-4 and an average grade of 6.2% copper over the initial 10-years of operations, resulting in estimated average annual copper production of 284,000 tonnes.
- Annual copper production is estimated at 366,000 tonnes in Year-4.
- Initial capital cost from July 2020, including contingency, is estimated at US$646M.
- Average total cash cost of US$1.16/lb of copper during the first 10-years.
- After-tax NPV, at an 8% discount rate, of US$5.5 billion.
- After-tax internal rate of return (IRR) of 77.0%, and a payback period of 2.3-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 2020 FS the Kakula mill would be constructed in two phases of 3.8 Mtpa each as the mining operations ramp-up to full production of 6.0 Mtpa, which leaves spare capacity in the mill of 1.6 Mtpa. The first module of 3.8 Mtpa commences production in Q3'21, and the second in Q1'23.

The life-of-mine (LOM) production scenario provides for 110.0 Mt to be mined at an average grade of 5.22% copper, producing 8.5 Mt of high-grade copper concentrate, containing approximately 10.8 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$5.5 billion. It has an after-tax IRR of 77.0% and a payback period of 2.3 years. The estimated initial capital cost, including contingency, is US$646M. The capital expenditure for off-site power, which is included in the initial capital cost, includes a US$36M advance payment to the DRC state-owned electricity company, SNEL, to upgrade Mwadingusha hydropower plants to provide the Kamoa-Kakula Project with access to clean electricity for its planned operations. Construction activities at Mwadingusha, which will supply 72 MW of power, are nearing completion. 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 2020 FS for a single 6.0 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 2020 FS 6.0 Mtpa mill feed and copper grade profile for the LOM are shown in Figure 1.8 and the concentrate and metal production for the LOM are shown in Figure 1.9.

Table 1.9 Kakula 2020 FS Results Summary

Item	Unit	Total
Total Processed		
Quantity Milled	kt	109,975
Copper Feed Grade	%	5.22
Total Concentrate Produced		
Copper Concentrate Produced	kt (dry)	8,542
Copper Concentrate	kt (dry)	8,542
Copper Recovery	%	85.23
Copper Concentrate Grade	%	57.32
Contained Copper in Concentrate	Mlb	10,795
Contained Copper in Concentrate	kt	4,897
Peak Annual Recovered Copper Production	kt	366
Ten Year Average		
Copper Concentrate Produced	kt (dry)	496
Contained Copper in Concentrate	kt	284
Mine-Site Cash Cost	US$/lb Payable Cu	0.52
Total Cash Cost	US$/lb Payable Cu	1.16
Key Financial Results		
Peak Funding	US$M	775
Initial Capital Costs	US$M	646
Expansion Capital Costs	US$M	594
Sustaining Capital Cost	US$M	1,265
Mine Site Cash Cost	US$/lb Payable Cu	0.62
Total Cash Costs After Credits	US$/lb Payable Cu	1.26
Site Operating Costs	US$/t Milled	58.73
After-Tax NPV8%	US$M	5,520
After-Tax IRR	%	77.0
Project Payback Period	Years	2.3
Project Life	Years	21

Table 1.10 Kakula 2020 FS Financial Results

	Discount Rate (%)	Before Taxation	After Taxation
Net Present Value (US$M)	Undiscounted	16,761	11,595
	4.0	11,258	7,832
	6.0	9,381	6,544
	8.0	7,892	5,520
	10.0	6,698	4,696
	12.0	5,729	4,024
Internal Rate of Return (%)	–	86.3	77.0
Project Payback Period (Years)	–	2.3	2.3

Table 1.11 Kakula 2020 FS Production and Processing

Item	Unit	Total LOM	Years 1–5	Years 1–10	LOM Average
Total Processed					
Quantity Milled	kt	109,975	4,638	5,345	5,237
Copper Feed Grade	%	5.22	6.56	6.21	5.22
Total Concentrate Produced					
Copper Concentrate Produced	kt (dry)	8,542	454	496	407
Copper Concentrate	kt (dry)	8,542	454	496	407
Copper Recovery	%	85.23	85.54	85.58	85.23
Copper Concentrate Grade	%	57.32	57.32	57.32	57.32
Contained Copper in Concentrate					
Copper	Mlb	10,795	574	626	514
Copper	kt	4,897	260	284	233
Payable Copper in Concentrate					
Copper	Mlb	10,444	555	606	497
Copper	kt	4,737	252	275	226
Payable Copper					
Copper	Mlb	10,444	555	606	497
Copper	kt	4,737	252	275	226

Figure 1.8 Kakula 2020 FS Process Production

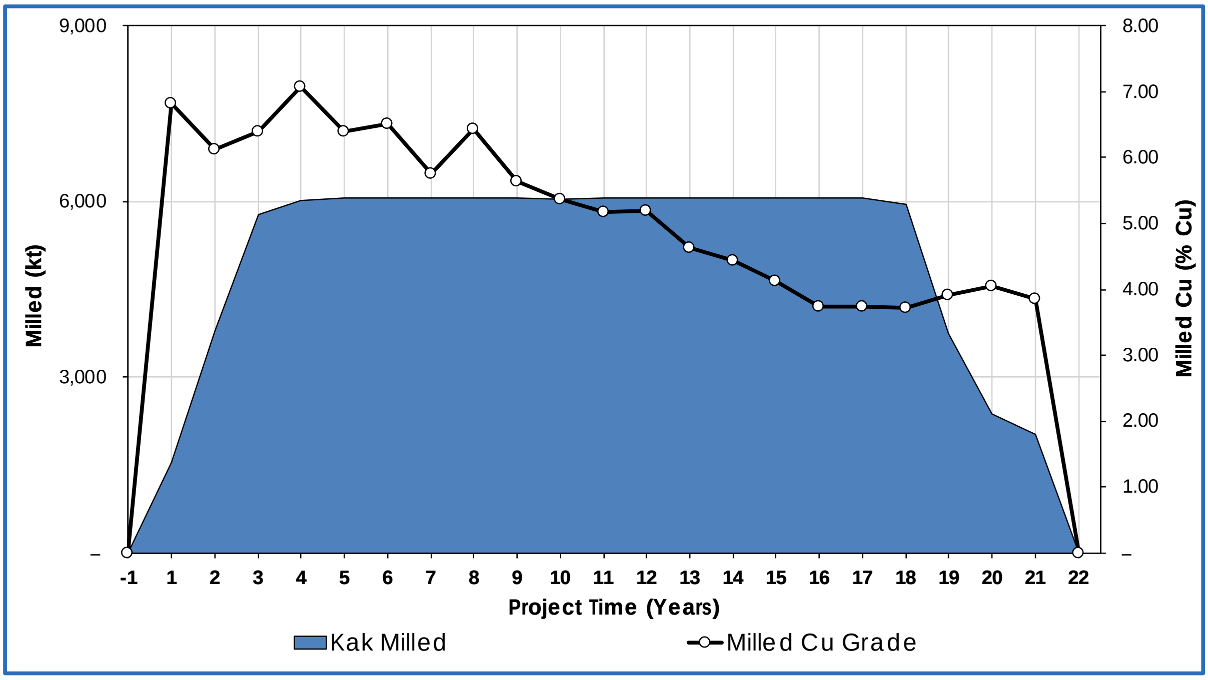


Figure by OreWin, 2020.

Figure 1.9 Kakula 2020 FS Concentrate and Metal Production

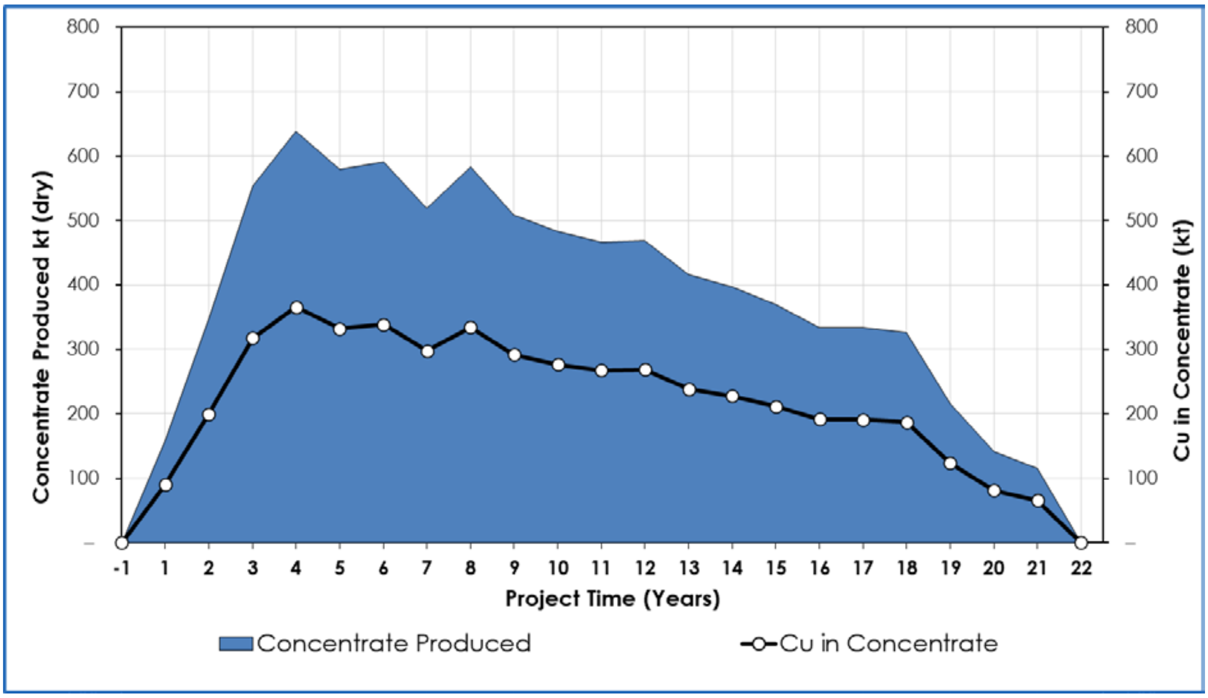


Figure by OreWin, 2020.

The annual and cumulative cash flows are shown in Figure 1.10 (annual cash flow is shown on the left vertical axis and cumulative cash flow on the right axis).

Figure 1.10 Kakula Mine Projected Cumulative Cash Flow

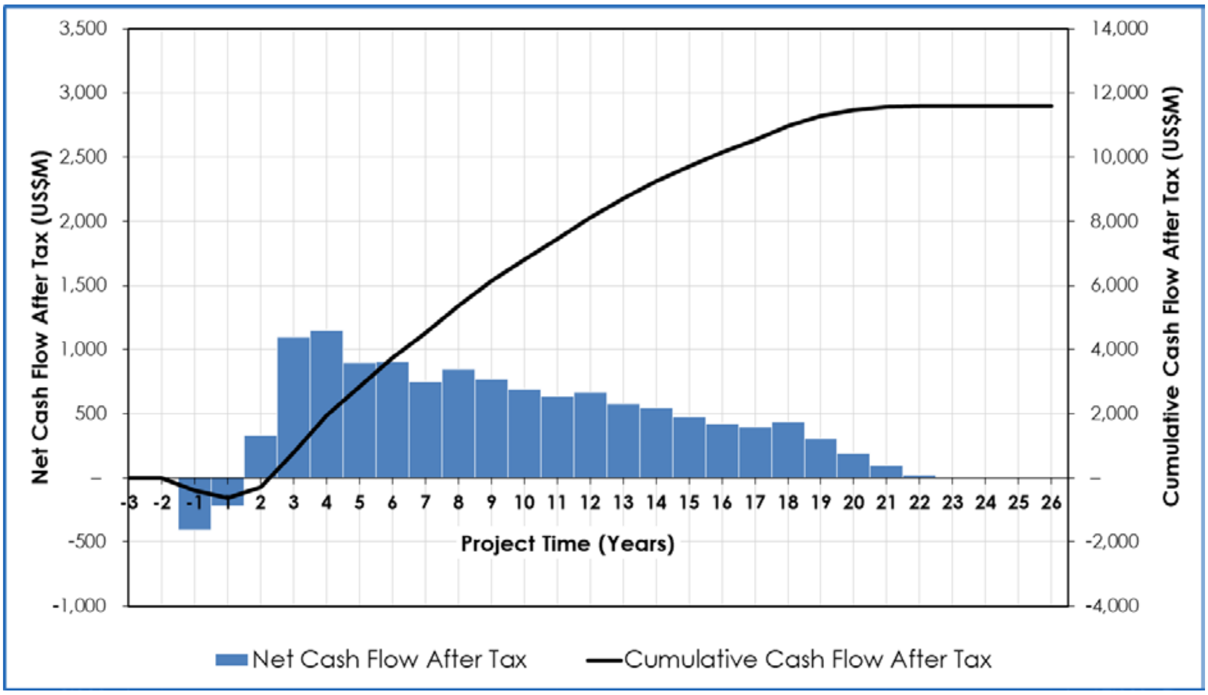


Figure by OreWin, 2020.

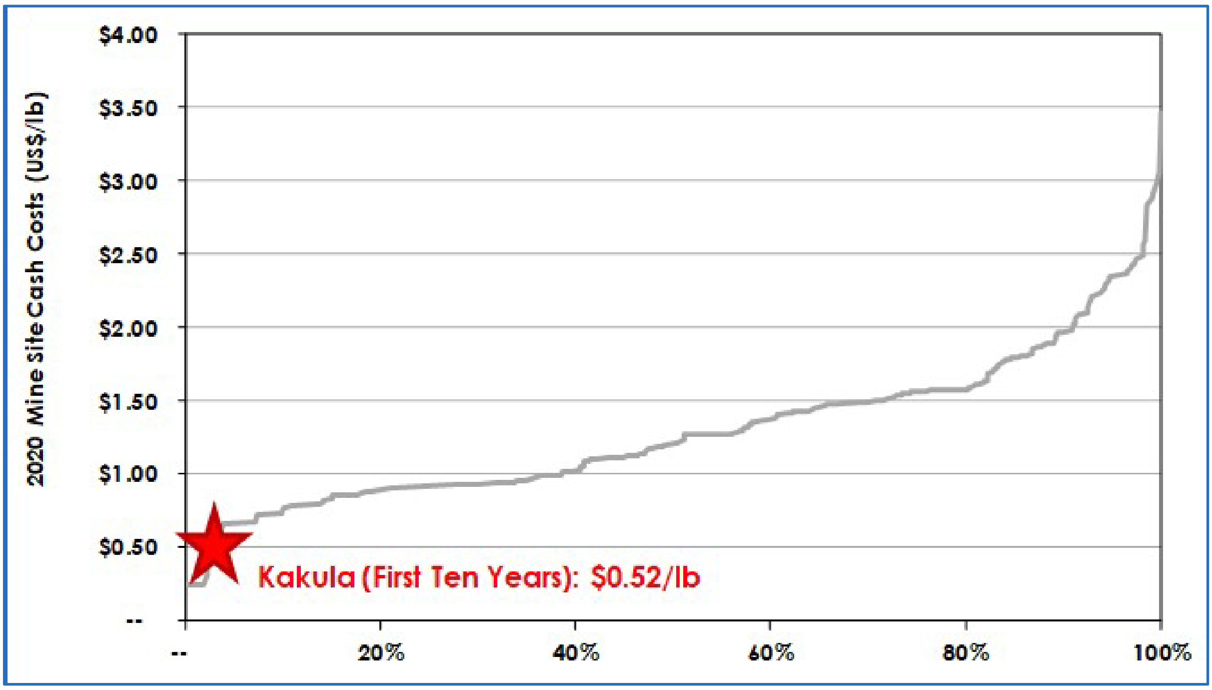
Figure 1.11 shows the average mine-site cash cost during the first 10-years of the Kakula 2020 FS on Wood Mackenzie's industry cost curve. This figure represents mine-site cash costs that reflect the direct cash costs of producing paid copper incorporating mining, processing and mine-site G&A costs.

Figure 1.12 shows the C1 pro-rata copper cash costs of the Kakula 2020 FS 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 realisation costs, having made appropriate allowance for the costs associated with the co-product revenue streams.

For both charts, the Kamoa-Kakula IDP20 was not reviewed by Wood Mackenzie prior to filing, and information was sourced from public disclosures.

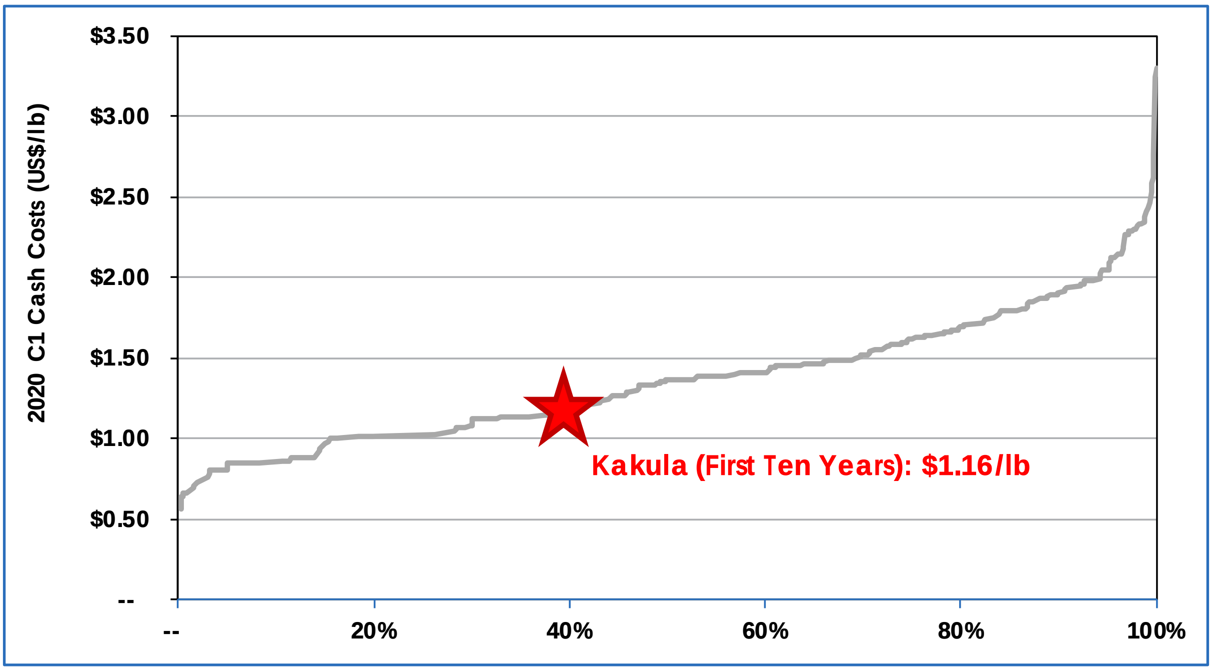
Table 1.12 summarises unit operating costs. Table 1.13 provides a breakdown of revenue and operating costs. Capital costs for the project are detailed in Table 1.14.

Figure 1.11 2020 Mine-Site Cash Costs (Includes All Operational Costs at Mine Site)



Source: Wood Mackenzie 2020

Figure 1.12 2020 C1 Copper Cash Costs



Source: Wood Mackenzie 2020.

Table 1.12 Kakula 2020 FS Unit Operating Costs

	Payable Cu (US$/lb)		
	Years 1–5	Years 1–10	LOM Average
Mine Site	0.48	0.52	0.62
Transport	0.32	0.32	0.32
Treatment and Refining Charges	0.11	0.11	0.11
Royalties and Export Tax	0.20	0.20	0.20
Total Cash Costs	1.12	1.16	1.26

Table 1.13 Kakula 2020 FS Revenue and Operating Costs

	Total LOM (US$M)	Years 1–5	Years 1–10	LOM Average
		(US$/t Milled)		
Revenue				
Copper in Concentrate	32,348	369.71	350.90	294.14
Gross Sales Revenue	32,348	369.71	350.90	294.14
Less: Realisation Costs				
Transport	3,383	38.76	36.73	30.77
Treatment and Refining	1,199	13.74	13.01	10.90
Royalties and Export Tax	2,106	24.10	22.85	19.15
Total Realisation Costs	6,689	76.60	72.59	60.82
Net Sales Revenue	25,660	293.12	278.31	233.32
Site Operating Costs				
Underground Mining	4,280	35.38	38.58	38.92
Processing	1,470	14.12	13.37	13.37
General and Administration	758	7.60	7.04	6.89
SNEL Discount	–294	–2.39	–2.55	–2.67
Customs Duties	245	2.13	2.21	2.23
Total	6,459	56.85	58.65	58.73
Net Operating Margin	19,201	236.27	219.66	174.59
Net Operating Margin (%)	74.83	80.61	78.93	74.83

Table 1.14 Kakula 2020 FS Capital Costs

Capital Costs (US$M)	Initial Capital (US$M)	Expansion Capital (US$M)	Sustaining Capital (US$M)	Total (US$M)
Underground Mining				
Underground Mining	131	202	538	871
Mining Infrastructure and Mobile Equipment	38	16	362	416
Capitalised Pre-Production	76	–	–	76
Subtotal	246	218	899	1,363
Off-site Power				
Power Supply Off Site	36	–	–	36
Subtotal	36	–	–	36
Concentrator and Tailings				
Plant	123	128	70	320
Tailings	13	26	88	127
Subtotal	136	154	157	448
Infrastructure				
Surface Infrastructure	69	101	14	184
Other Infrastructure	–	–	–	–
Subtotal	69	101	14	184
Indirects				
EPCM	35	17	0	53
Owners Cost	66	47	–	114
Customs Duties	8	18	40	66
Closure	–	–	82	82
Subtotal	110	83	122	315
Capital Expenditure Before Contingency	596	556	1,193	2,346
Contingency	50	38	72	159
Capital Expenditure After Contingency	646	594	1,265	2,505

Totals have been rounded.

Table 1.15 Kakula 2020 FS 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**	**4.50**
Undiscounted	4,225	7,519	10,911	11,595	14,353	17,532	19,928
4.0%	2,828	5,072	7,370	7,832	9,704	11,852	13,457
6.0%	2,334	4,227	6,156	6,544	8,117	9,918	11,256
8.0%	1,935	3,551	5,190	5,520	6,857	8,384	9,513
10.0%	1,609	3,005	4,413	4,696	5,845	7,153	8,116
12.0%	1,340	2,558	3,779	4,024	5,022	6,154	6,982
15.0%	1,018	2,028	3,031	3,232	4,052	4,977	5,649
IRR (%)	38.5	57.9	74.0	77.0	88.9	100.4	106.9

1.17 Kakula-Kansoko 2020 PFS

The Kakula-Kansoko 2020 PFS is a combined schedule comprising of the Kakula 2020 FS (6.0 Mtpa) and a Kansoko 1.6 Mtpa schedule. The mine design for Kansoko is the same as what had been previously called the Kamoa 2019 PFS. Costs were updated using the Kakula 2020 costs as a basis. Ivanhoe has developed twin declines at the Kansoko Mine on the Kansoko areas of the Kamoa deposit. 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 2019 PFS.

1.17.1 Kakula-Kansoko 2020 PFS Mining

A probable combined Mineral Reserve of approximately 235.2 Mt grading at 4.47% Cu has been defined in the Kakula and Kansoko mines. The overall mining rate is 7.6 Mtpa, constraining Kansoko to 1.6 Mtpa until Kakula is depleted, at which point the mining rate at Kansoko increases to reducing to 6.0 Mtpa for the remainder of the mine life after the completion of Kakula. The mine design and schedule for Kakula is that of the Kakula 2020 FS. The Kansoko ore zones occur at depths ranging from approximately 60–1,235 m. Access to the mine will be via twin declines. Main declines and ventilation raises are shown in Figure 1.13. 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 1.13 Kakula-Kansoko 2020 PFS Kansoko Mine Access and Ventilation

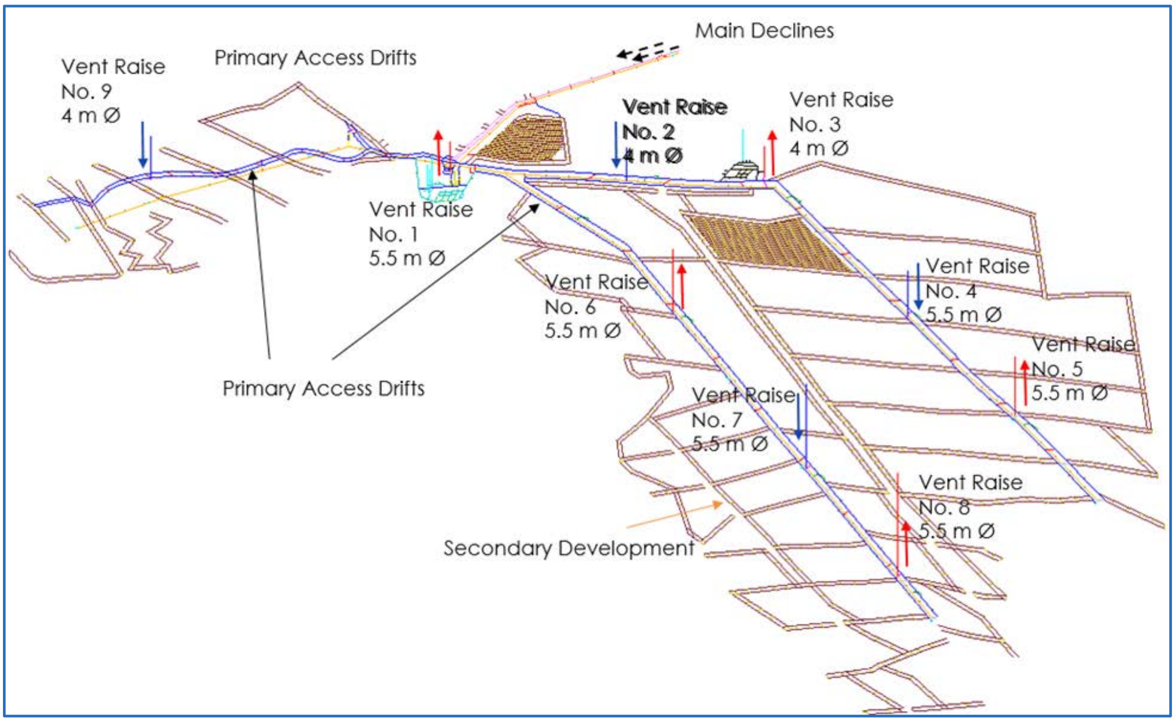


Figure by Stantec, 2017.

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 m. 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 Kansoko development schedule initially is 1.6 Mtpa creating a combined 7.6 Mtpa for the Kakula-Kansoko 2020 PFS. The Kansoko schedule then focuses on establishment of necessary mine services and support infrastructure to set up the initial production mining areas and ramp-up to 6.0 Mtpa ore production and associated development waste.

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 1.16 shows the Kansoko LOM production summary.

Table 1.16 Kansoko LOM Production Summary

Production by Mining Method	Mined (kt)	Metres (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.

Kansoko 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.

1.17.2 Kakula-Kansoko 2020 PFS Process

The Kakula-Kansoko 2020 PFS process plant is similar to the Kakula flow sheet and consists of a 7.6 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. ROM material from Kansoko will be trucked to Kakula, where it will be introduced to the circuit via the bulk reclaim tip system at a rate of 1.6 Mtpa.

The Kakula-Kansoko concentrator design incorporates a run-of-mine stockpile, followed by cone crushers operating in closed circuit with vibrating screens to produce 100% passing 50 mm material that is stockpiled. The crushed ore is fed to the High Pressure Grinding Rolls (HPGR) operating in closed circuit with wet screening, at a product size of 80% (P_{80}) passing 4.5 mm which is gravity fed to the milling circuit. The milling circuit incorporates two stages of ball milling in series in closed circuit with cyclone clusters for further size reduction and classification to a target grind size of 80% passing 53 µm. The milled slurry is pumped to the rougher and scavenger flotation circuit where the high-grade, or fast-floating rougher concentrate, and medium-grade, or slow-floating scavenger concentrate, are separated for further upgrading. The rougher concentrate is upgraded in the low entrainment high-grade cleaner stage to produce a high-grade concentrate. The medium-grade or scavenger concentrate together with the tailings from the high-grade cleaner stage and the recycled scavenger recleaner tailings are combined and further upgraded in the scavenger cleaner circuit.

The concentrate produced from the scavenger cleaner circuit, representing roughly 15% of the mill feed, is re-ground to a P_{80} of 10 µm prior to final cleaning in the low entrainment scavenger recleaner stage. The scavenger recleaner concentrate is then combined with the high-grade cleaner concentrate to form final concentrate. The final concentrate is then thickened and pumped to the concentrate filter. Final filtered concentrate is then bagged for shipment to market.

The scavenger tailings and scavenger cleaner tailings are combined and thickened prior to being pumped to the backfill plant and/or to the tailings storage facility. Backfill utilizes about half of tailings and the remainder will be pumped to the tailings storage facility.

1.17.3 Kakula-Kansoko 2020 PFS Transport

A phased logistics solution is proposed in the Kakula-Kansoko 2020 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.

1.17.4 Kakula-Kansoko 2020 PFS Results

The Kakula-Kansoko 2020 PFS is a combined schedule comprising of the Kakula 2020 FS (6.0 Mtpa) and a Kansoko 1.6 Mtpa schedule. The Kakula-Kansoko 2020 PFS maintains the development of the Kakula Deposit as a stand-alone 6.0 Mtpa mining and combined 7.6 Mtpa processing complex. The combined LOM production scenario schedules 235.2 Mt to be mined at an average grade of 4.47% copper, producing 20.0 Mt of high-grade copper concentrate, containing approximately 20 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$6.6 billion. The Kakula-Kansoko 2020 PFS has an after-tax IRR of 69.0% and a payback period of 2.5 years. The LOM average mine site cash cost is US$0.64/lb of copper.

The estimated initial capital cost, including contingency, is US$695M. The key results of the Kakula-Kansoko 2020 PFS are summarised in Table 1.17. Table 1.18 summarises the financial results.

While production is at the 1.6 Mtpa rate ore will be moved to surface with haulage trucks and hauled to the Kakula processing facility. Once the production ramps up to 6.0 Mtpa ore will be moved to the surface via conveyor and transferred on to an overland conveyor to the Kakula processing facility.

Table 1.17 Kakula-Kansoko 2020 PFS Summary

Item	Unit	Total
Total Processed		
Quantity Milled	kt	235,157
Copper Feed Grade	%	4.47
Total Concentrate Produced		
Copper Concentrate Produced	kt (dry)	19,948
Copper Recovery	%	86.27
Copper Concentrate Grade	%	45.49
Contained Copper in Concentrate	Mlb	20,006
Contained Copper in Concentrate	kt	9,075
Peak Annual Contained Metal in Concentrate	kt	427
10-Year Average		
Copper Concentrate Produced	kt (dry)	622
Contained Copper in Concentrate	kt	331
Mine Site Cash Cost	US$/lb	0.55
Total Cash Cost	US$/lb	1.23
Key Financial Results		
Peak Funding	US$M	848
Initial Capital Cost	US$M	695
Expansion Capital Cost	US$M	750
Sustaining Capital Cost	US$M	2,827
LOM Average Mine Site Cash Cost	US$/lb Cu	0.64
LOM Average Total Cash Cost	US$/lb Cu	1.44
Site Operating Cost	US$/t Milled	52.95
After-Tax NPV8%	US$M	6,604
After-Tax IRR	%	69.0
Project Payback Period	Years	2.5
Project Life	Years	37

Table 1.18 Kakula-Kansoko 2020 PFS Financial Results

	Discount Rate (%)	Before Taxation	After Taxation
Net Present Value (US$M)	Undiscounted	27,805	18,373
	4.0	15,562	10,422
	6.0	12,179	8,204
	8.0	9,757	6,604
	10.0	7,967	5,415
	12.0	6,608	4,505
Internal Rate of Return (%)	–	78.5	69.0
Project Payback Period (Years)	–	2.5	2.5

Table 1.19 Kakula-Kansoko 2020 PFS Production and Processing

Item	Unit	Total LOM	Years 1–5	Years 1–10	LOM Average
Total Processed					
Quantity Milled	kt	235,157	5,536	6,568	6,356
Copper Feed Grade	%	4.47	6.20	5.87	4.47
Total Concentrate Produced					
Copper Concentrate Produced	kt (dry)	19,948	542	622	539
Copper Concentrate	kt (dry)	19,948	542	622	539
Copper Recovery	%	86.27	85.61	85.84	86.27
Copper Concentrate Grade	%	45.49	54.21	53.27	45.49
Contained Copper in Concentrate					
Copper	Mlb	20,006	648	730	541
Copper	kt	9,075	294	331	245
Payable Copper in Concentrate					
Copper	Mlb	19,356	627	706	523
Copper	kt	8,780	284	320	237
Payable Copper					
Copper	Mlb	19,356	627	706	523
Copper	kt	8,780	284	320	237

Table 1.20 Kakula-Kansoko 2020 PFS Capital Costs

Capital Costs (US$M)	Initial Capital (US$M)	Expansion Capital (US$M)		Sustaining Capital (US$M)	Total (US$M)
		Kakula 6.0 Mtpa / 7.6 Mtpa Plant	Kansoko to 1.6 Mtpa		
Underground Mining					
Underground Mining	158	202	97	1,068	1,525
Capitalised Pre-Production	76	–	–	–	76
Mining Mobile Equipment	55	43	17	922	1,036
Subtotal	289	245	114	1,990	2,638
Off-site Power					
Power Supply Off Site	36	–	–	–	36
Subtotal	36	–	–	–	36
Concentrator and Tailings					
Plant	123	128	–	135	386
Tailings	13	12	–	240	265
Subtotal	136	139	–	375	651
Infrastructure					
Plant Infrastructure	69	101	–	14	184
Conveyor Kansoko to Kakula	–	–	–	95	95
Subtotal	69	101	–	109	279
Indirects					
EPCM	37	15	9	0	62
Owners Cost	67	47	4	–	117
Customs Duties	8	23	0	89	120
Closure	–	–	–	81	81
Subtotal	113	85	13	170	380
Capital Expenditure Before Contingency	642	571	126	2,644	3,984
Contingency	52	41	12	183	288
Capital Expenditure After Contingency	695	612	139	2,827	4,272

Figure 1.14 Kakula-Kansoko 2020 PFS Process Production

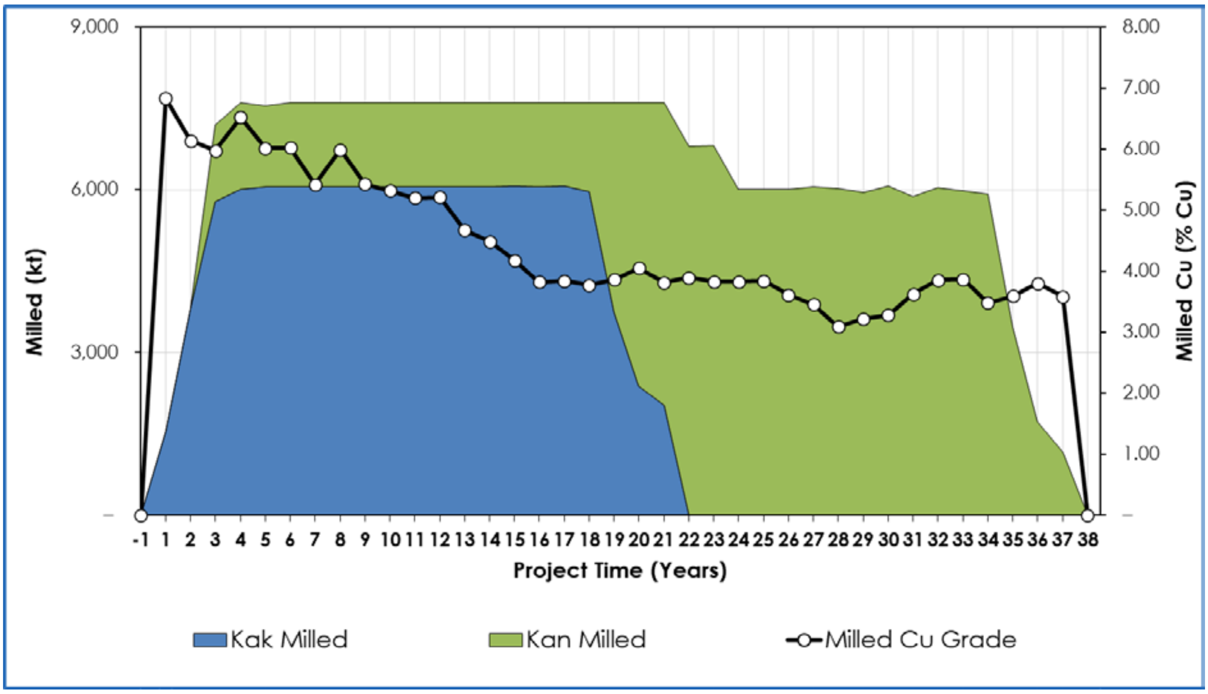


Figure by OreWin, 2020.

Figure 1.15 Kakula-Kansoko 2020 PFS Concentrate and Metal Production

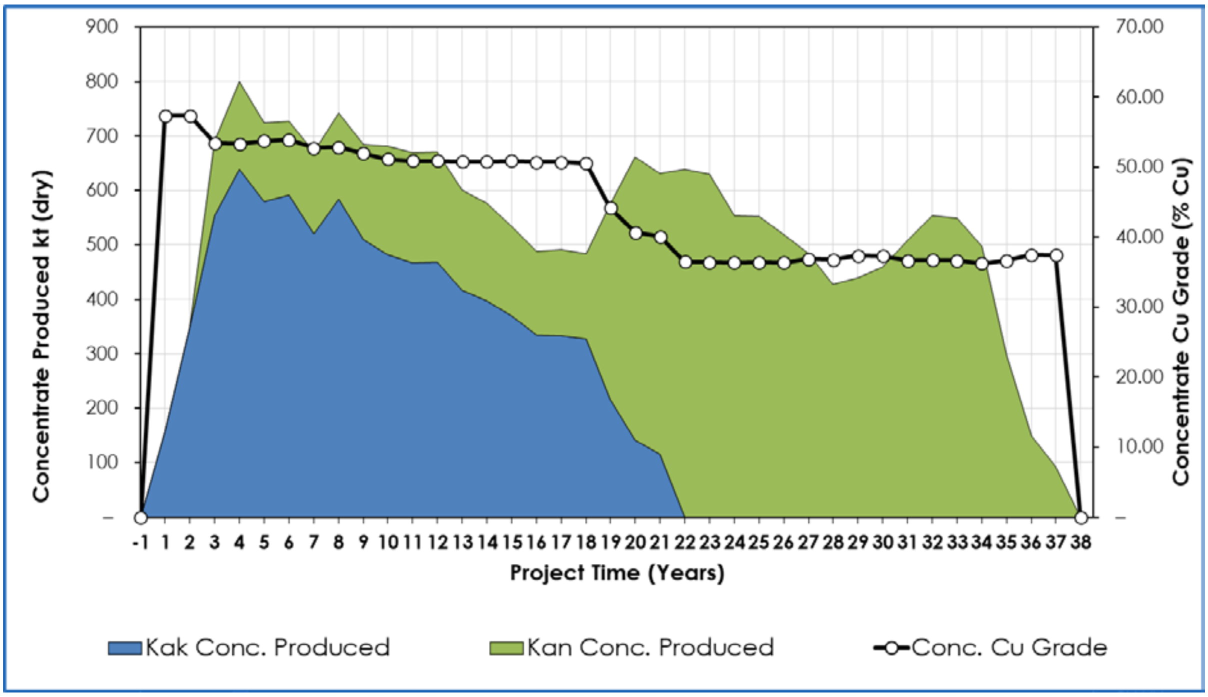


Figure by OreWin, 2020.

The annual and cumulative cash flows are shown in Figure 1.16 (annual cash flow is shown on the left vertical axis and cumulative cash flow on the right axis).

Figure 1.16 Kakula-Kansoko 2020 PFS Mine Projected Cumulative Cash Flow

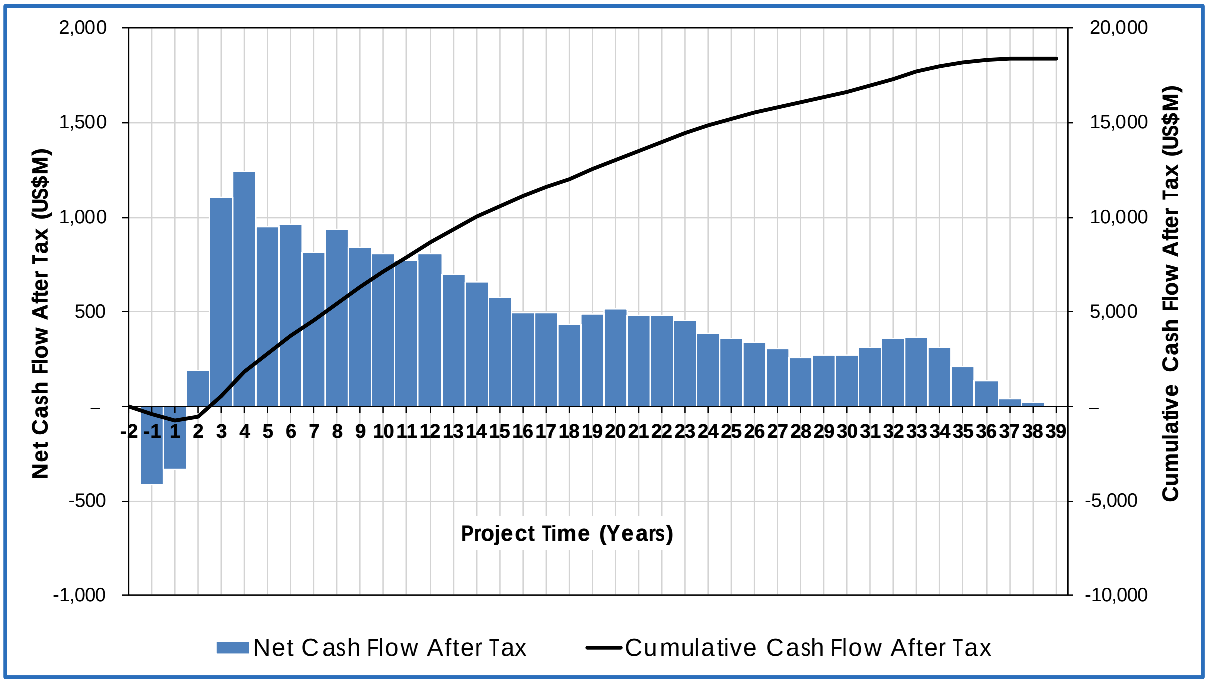


Figure by OreWin, 2020.

Table 1.21 Kakula-Kansoko 2020 PFS 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**	**4.50**
Undiscounted	4,758	10,753	17,101	18,373	23,487	29,393	33,873
4.0%	2,847	6,181	9,714	10,422	13,275	16,560	19,031
6.0%	2,241	4,871	7,648	8,204	10,448	13,026	14,957
8.0%	1,774	3,911	6,156	6,604	8,419	10,498	12,047
10.0%	1,409	3,188	5,044	5,415	6,915	8,631	9,902
12.0%	1,117	2,629	4,194	4,505	5,770	7,212	8,274
15.0%	781	2,000	3,247	3,495	4,501	5,644	6,480
IRR (%)	29.5	49.2	66.0	69.0	81.0	93.0	99.8

Figure 1.17 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. Kakula-Kansoko is based on the capital costs incurred in 2019, the capital costs incurred in the six months ended 30 June 2020 and the estimated initial and expansion capital costs from 1 July 2020 in the Kakula-Kansoko 2020 PFS. Kakula-Kansoko's first 10 years' average annual production of copper in concentrate are considered to be its nominal copper production. The Kamoa-Kakula IDP20 was not reviewed by Wood Mackenzie prior to filing.

Figure 1.17 Capital Intensity for Large-Scale Copper Projects

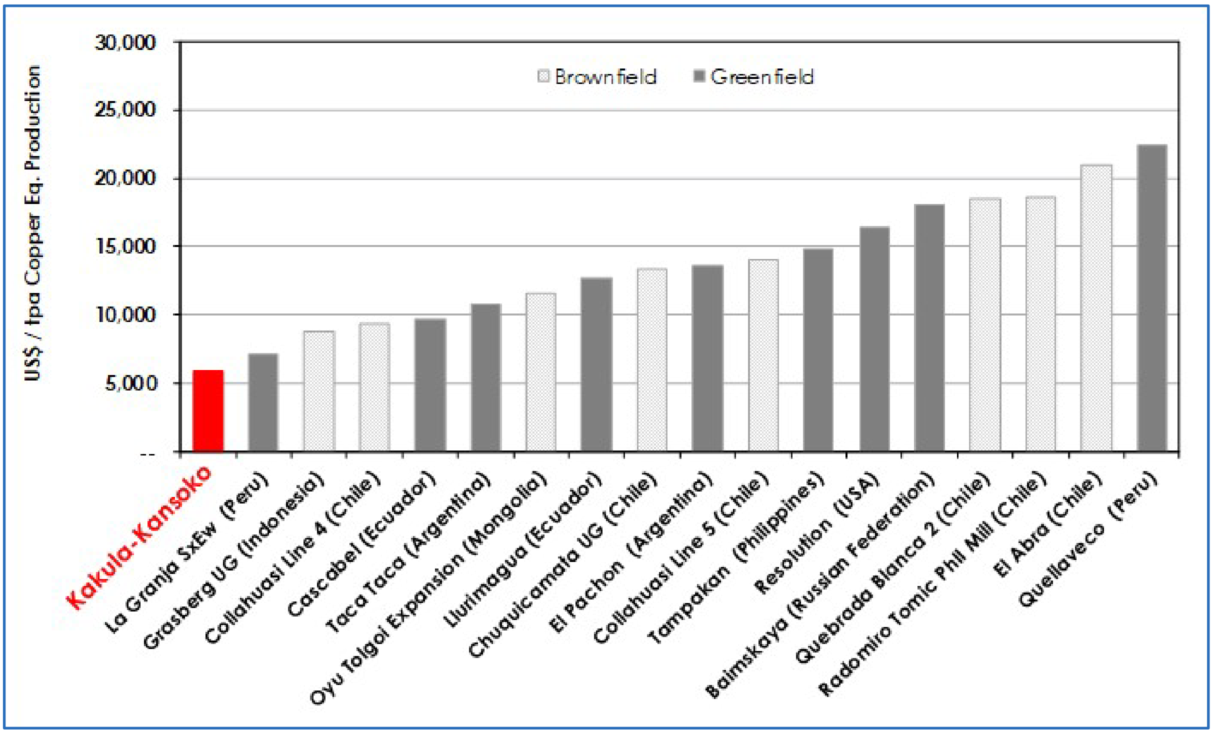


Figure by Ivanhoe, 2020. Source: Wood Mackenzie.

1.18 Kamoa-Kakula 2020 PEA

The Kamoa-Kakula 2020 PEA analyses a production case with an expansion of the Kakula concentrator processing facilities, and associated infrastructure to 19 Mtpa and includes a smelter and eight separate underground mining operations with associated capital and operating costs. The locations of the eight mines and the boundaries for the FS, PFS and PEA cases are shown in Figure 1.18. The details of the Kamoa-Kakula 2020 PEA are in Section 24. The eight mines ranked by their relative net present values are:

- Kakula Mine (FS 6.0 Mtpa).
- Kansoko Mine (PFS 1.6 Mtpa to 6.0 Mtpa).
- Kakula West Mine (PEA 6.0 Mtpa).
- Kamoa North Mine 1 (PEA 6.0 Mtpa).
- Kamoa North Mine 2 (PEA 6.0 Mtpa).
- Kamoa North Mine 3 (PEA 6.0 Mtpa).
- Kamoa North Mine 4 (PEA 3.0 Mtpa).
- Kamoa North Mine 5 (PEA 1.0 Mtpa).

Figure 1.18 Kamoa-Kakula IDP20 Mining Locations

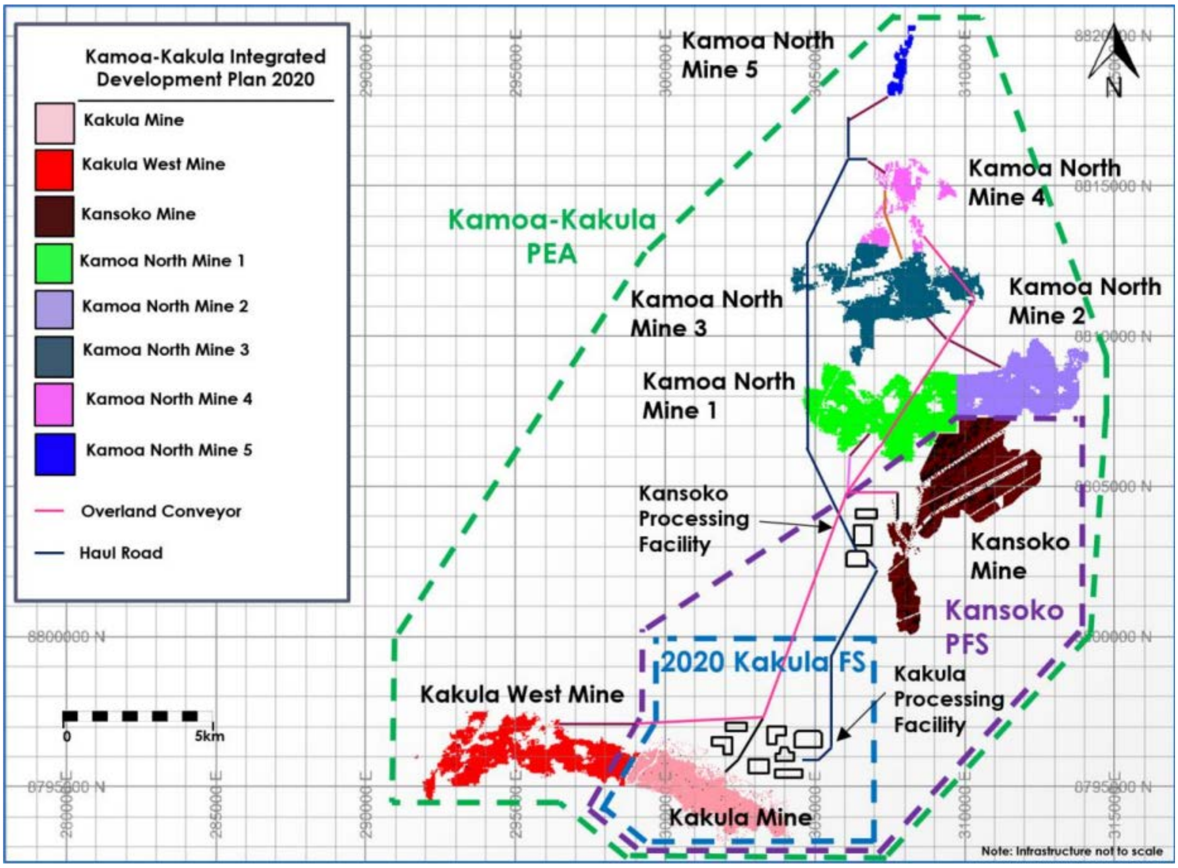


Figure by OreWin, 2020.

The Kamoa-Kakula 2020 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 potential development scenarios at the Kamoa-Kakula Project include the Kamoa-Kakula IDP20 development scenario shown in Figure 1.19.

Figure 1.19 Kamoa-Kakula IDP20 Long-Term Development Scenario

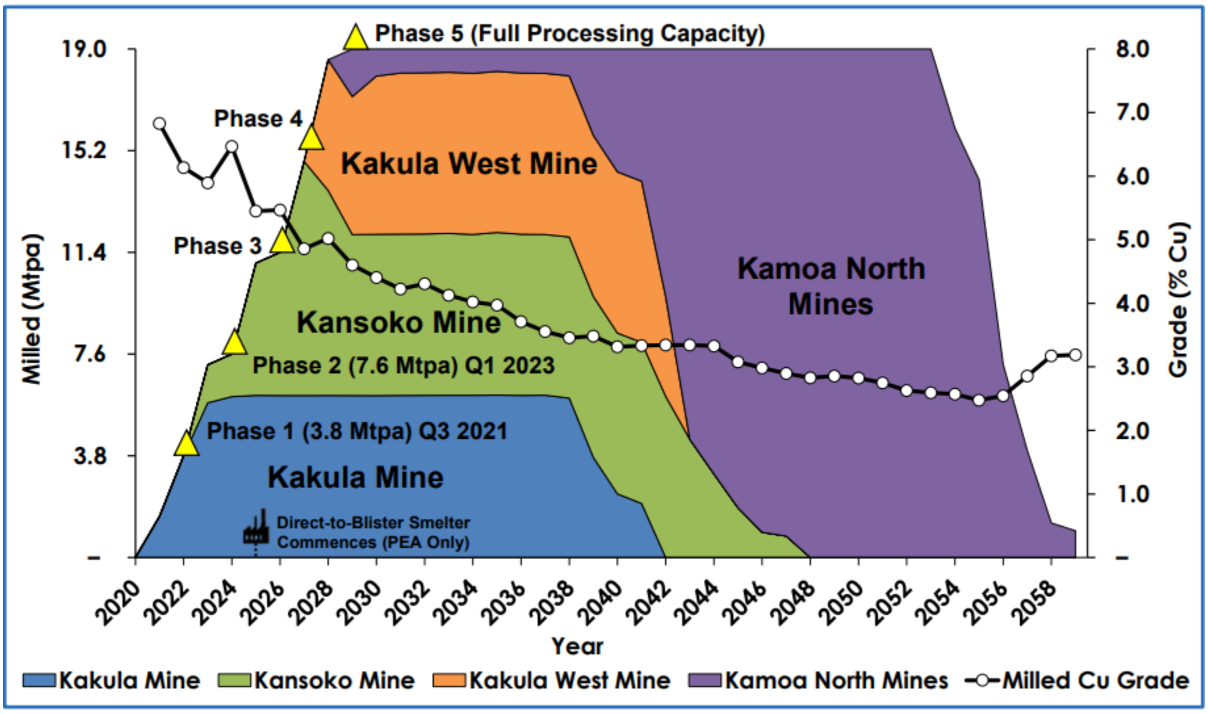


Figure by OreWin, 2020.

A site plan showing the locations of the mines and key infrastructure for Kakula and Kansoko mines is shown in Figure 1.20.

The Kamoa-Kakula 2020 PEA as part of the Kamoa-Kakula IDP20 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 2020 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 2020 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. 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 1.20 Kamoa-Kakula IDP20 Site Plan

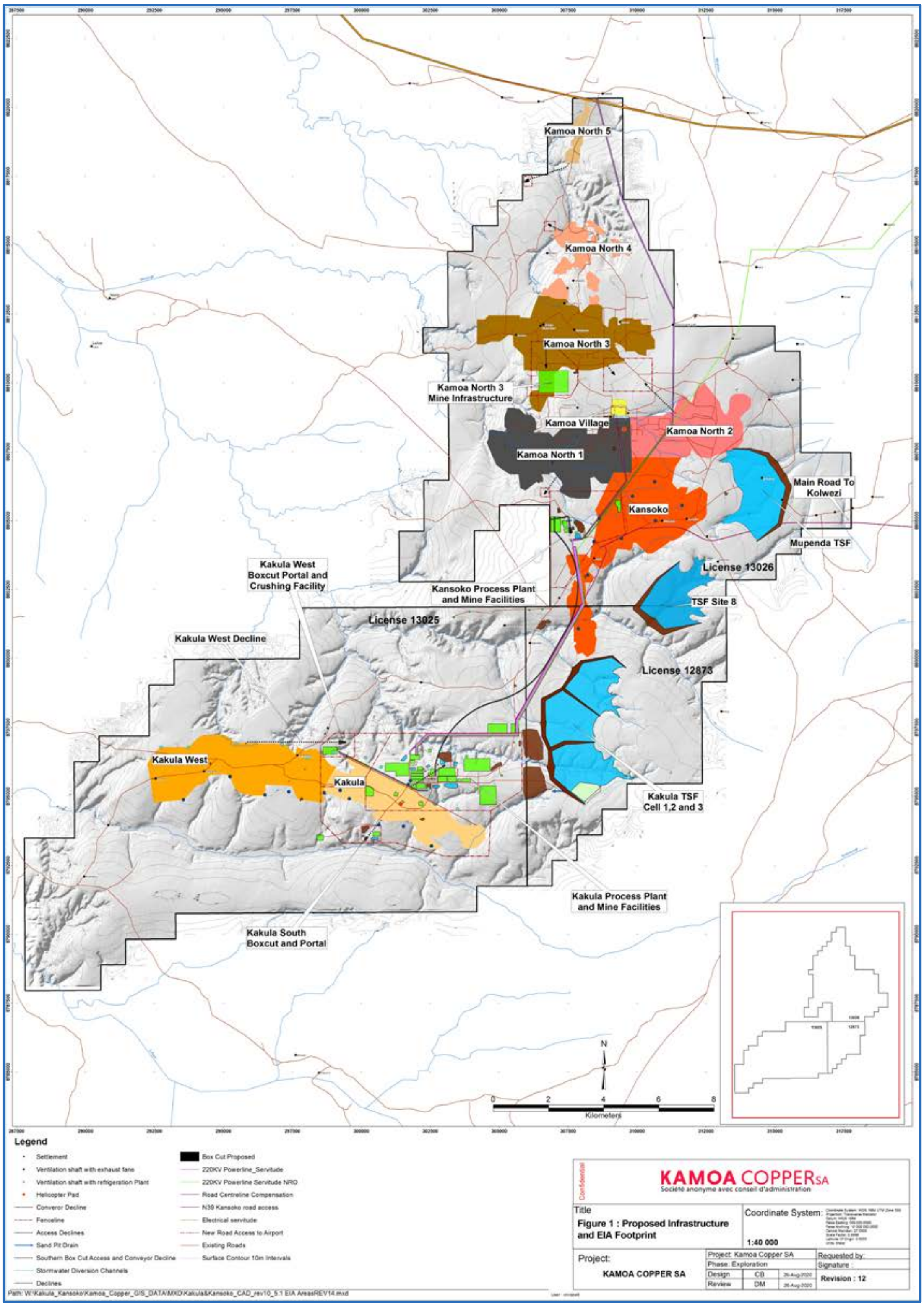


Figure by Kamoa Copper SA, 2020.

1.18.1 Kamoa-Kakula 2020 PEA Results Summary

The Kamoa-Kakula 2020 PEA assesses an alternative development option of mining several deposits on the Kamoa-Kakula Project as an integrated, 19 Mtpa mining, processing and smelting complex, built in three stages. This scenario envisages the construction and operation of eight separate mines: first, an initial 6.0 Mtpa mining operation would be established at the Kakula Mine on the Kakula Deposit; this is followed by a subsequent, separate 6.0 Mtpa mining operation at the Kansoko Mine using the existing twin declines that were completed in 2017; a third 6.0 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 five other mines in the Kamoa North area to maintain throughput of 19 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 and Kansoko. Included in this scenario is the construction of a direct-to-blister flash (DBF) copper smelter with a capacity of one million tonnes of copper concentrate per annum.

A summary of the key results for the Kamoa-Kakula 2020 PEA scenario are:

- Very-high-grade initial phase of production is projected to have a grade of 6.8% copper in the first year of production and an average grade of 5.13% copper over the initial 10-years of operations, resulting in estimated average annual copper production of 501,000 tonnes.
- Initial capital cost, including contingency, is estimated at US$715M.
- Average total cash cost of US$1.07/lb of copper during the first 10-years, including sulfuric acid credits.
- After-tax NPV, at an 8% discount rate, of US$11.12 billion.
- After-tax internal rate of return (IRR) of 56.2%, and a payback period of 3.6-years.

The estimated initial capital cost, including contingency, is US$715M.

The Kamoa-Kakula 2020 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 1.22 summarises the financial results. Key results of the Kamoa-Kakula 2020 PEA are summarised in and Table 1.23. The mining production statistics are shown in Table 1.24. The Kamoa-Kakula 2020 PEA 19 Mtpa mill feed and copper grade profile for the life-of-mine (LOM) are shown in Figure 1.21 and the concentrate and metal production for the LOM are shown in Figure 1.22. Table 1.25 summarises unit operating costs.

Table 1.22 Kamoa-Kakula 2020 PEA Financial Results

	Discount Rate	Before Taxation	After Taxation
Net Present Value (US$M)	Undiscounted	56,564	37,844
	4.0%	29,097	19,416
	6.0%	21,816	14,520
	8.0%	16,756	11,117
	10.0%	13,136	8,681
	12.0%	10,476	6,891
Internal Rate of Return (%)	–	66.4	56.2
Project Payback Period (Years)	–	3.4	3.6

Table 1.23 Kamoa-Kakula 2020 PEA Results Summary for 19 Mtpa Production

Item	Unit	Total
Total Processed		
Quantity Milled	kt	597,621
Copper Feed Grade	%	3.63
Total Concentrate Produced		
Copper Concentrate Produced	kt (dry)	42,818
Copper Recovery	%	86.42
Copper Concentrate Grade	%	43.76
Contained Copper in Concentrate - External Smelter	Mlb	13,251
Contained Copper in Concentrate - External Smelter	kt	6,010
Contained Copper in Blister - Internal Smelter	Mlb	27,641
Contained Copper in Blister - Internal Smelter	kt	12,538
Peak Annual Recovered Copper Production	kt	805
10-Year Average		
Copper Concentrate Produced	kt (dry)	1,043
Contained Copper in Conc. - External Smelter	kt	248
Contained Copper in Blister - Internal Smelter	kt	253
Mine-Site Cash Cost	US$/lb Cu	0.65
Total Cash Cost	US$/lb Cu	1.07
Key Financial Results		
Peak Funding	US$M	784
Initial Capital Cost	US$M	715
Expansion Capital Cost	US$M	4,461
Sustaining Capital Cost	US$M	11,958
LOM Average Mine Site Cash Cost	US$/lb Cu	0.92
LOM Average Total Cash Cost	US$/lb Cu	1.28
Site Operating Cost	US$/t Milled	62.44
After-Tax NPV8%	US$M	11,117
After-Tax IRR	%	56.2
Project Payback Period	Years	3.6
Project Life	Years	43

Table 1.24 Kamoa-Kakula 2020 PEA Production and Processing

Item	Unit	Total LOM	Years 1–5	Years 1–10	LOM Average
Total Processed					
Quantity Milled	kt	597,621	6,227	11,394	13,898
Copper Feed Grade	%	3.63	5.95	5.13	3.63
Total Concentrate Produced					
Copper Concentrate Produced	kt (dry)	42,818	613	1,043	996
Copper Concentrate - External Smelter	kt (dry)	11,944	413	443	278
Copper Concentrate - Internal Smelter	kt (dry)	30,874	200	600	718
Copper Recovery	%	86.42	85.98	86.44	86.42
Copper Concentrate Grade	%	43.76	52.02	48.38	43.76
Contained Copper in Concentrate - External Smelter					
Copper	Mlb	13,251	496	548	308
Copper	kt	6,010	225	248	140
Payable Copper in Concentrate - External Smelter					
Copper	Mlb	12,820	480	530	298
Copper	kt	5,815	218	240	135
Contained Copper in Blister - Internal Smelter					
Copper	Mlb	27,641	204	557	643
Copper	kt	12,538	92	253	292
Payable Copper in Blister - Internal Smelter					
Copper	Mlb	27,558	203	555	641
Copper	kt	12,500	92	252	291
Payable Copper					
Copper	Mlb	40,378	683	1,085	939
Copper	kt	18,315	310	492	426

Table 1.25 Kamoa-Kakula 2020 PEA Unit Operating Costs

Item	Payable Copper (US$/lb)		
	Years 1–5	Years 1–10	LOM Average
Mine Site	0.56	0.57	0.81
Smelter	0.04	0.08	0.11
Transport	0.29	0.24	0.23
Treatment and Refining Charges	0.10	0.08	0.07
Royalties and Export Tax	0.19	0.18	0.17
Total Cash Costs	1.18	1.15	1.40
Sulfuric Acid Credits	0.04	0.09	0.12
Total Cash Costs After Credits	1.14	1.07	1.28

Figure 1.21 Kamoa-Kakula 2020 PEA Process Production

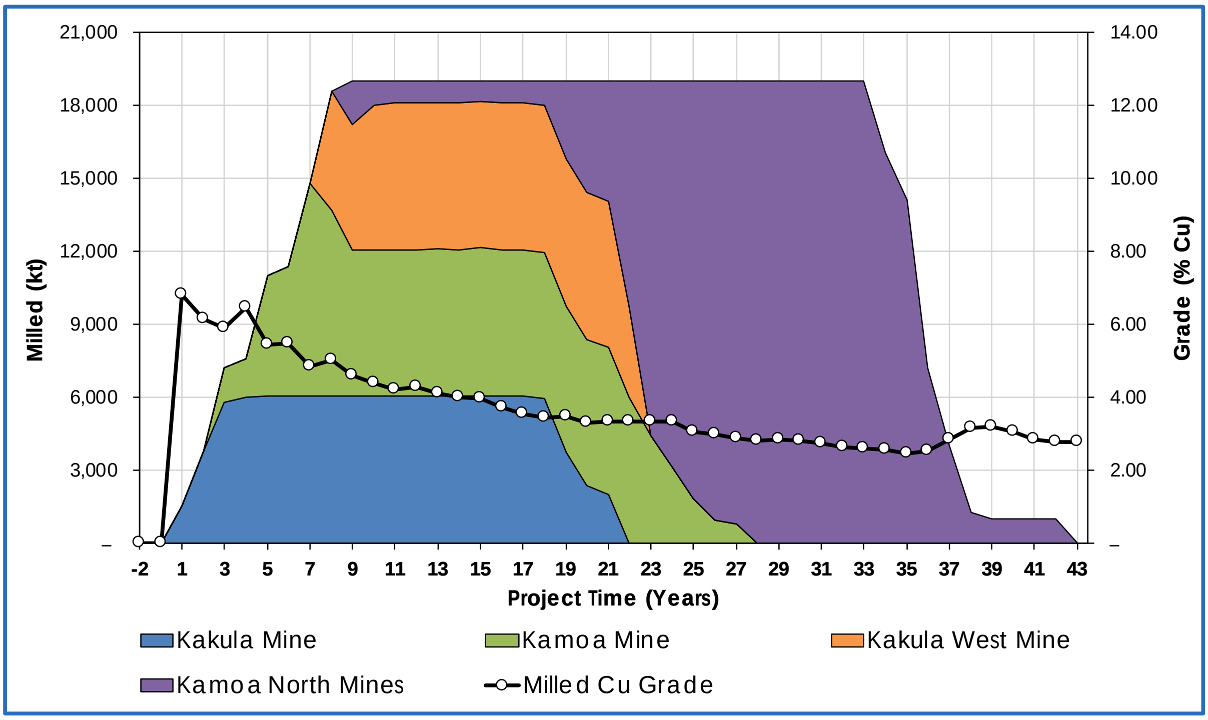


Figure by OreWin, 2020.

Figure 1.22 Kamoa-Kakula 2020 PEA Concentrate and Metal Production

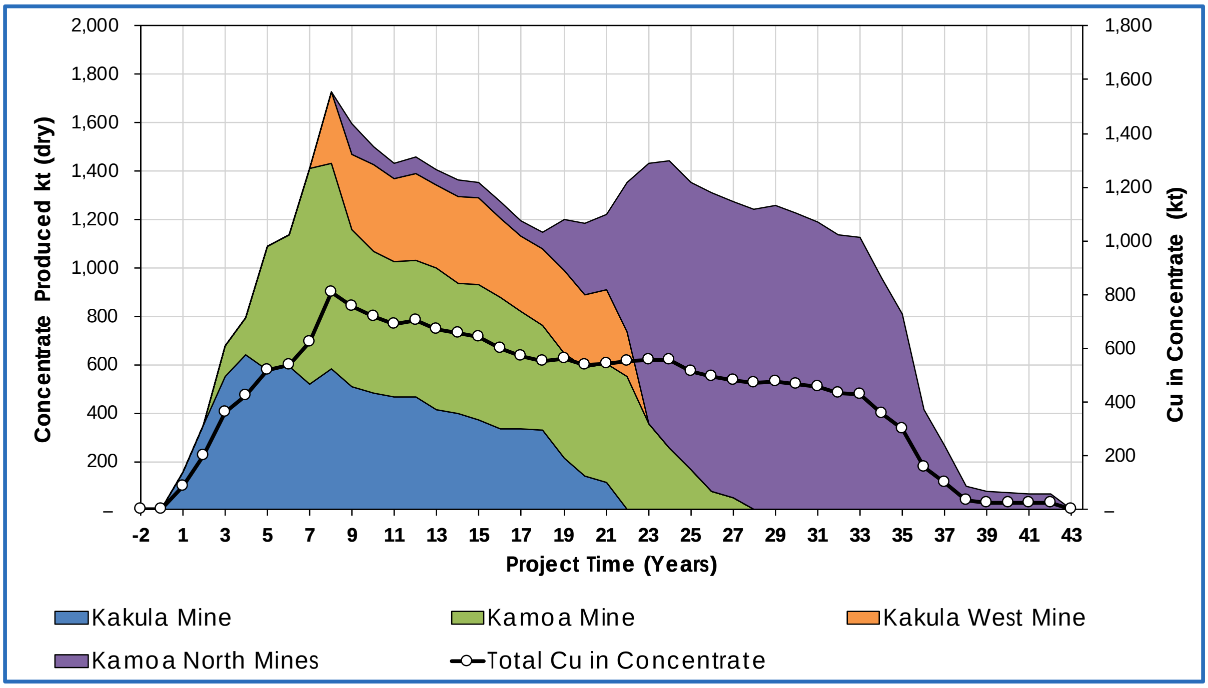


Figure by OreWin, 2020.

Table 1.26 provides a breakdown of revenue and operating costs. Capital costs for the project are detailed in Table 1.27.

Table 1.26 Kamoa-Kakula 2020 PEA Revenue and Operating Costs

	Total LOM (US$M)	Years 1–5	Years 1–10	LOM Average
		(US$/t) Milled		
Revenue				
Copper in Blister	85,430	101.06	151.08	142.95
Copper in Concentrate	39,713	237.97	143.89	66.45
Acid Production	4,930	4.38	8.23	8.25
Gross Sales Revenue	130,074	343.41	303.20	217.65
Less: Realisation Costs				
Transport	9,132	31.62	23.03	15.28
Treatment and Refining	2,941	10.68	7.83	4.92
Royalties and Export Tax	7,038	21.00	16.74	11.78
Total Realisation Costs	19,111	63.30	47.60	31.98
Net Sales Revenue	110,963	280.11	255.60	185.67
Site Operating Costs				
Underground Mining	21,070	39.57	35.26	35.26
Processing	8,811	13.82	14.02	14.74
Tailings	67	0.26	0.15	0.11
Smelter	4,464	4.58	7.48	7.47
General and Administration	1,767	8.30	5.28	2.96
SNEL Discount	–292	–3.43	–2.57	–0.49
Customs Duties	1,426	2.55	2.48	2.39
Total	37,313	65.66	62.10	62.44
Net Operating Margin	73,650	214.45	193.50	123.24
Net Operating Margin (%)	66.37	76.56	75.71	66.37

Totals have been rounded.

Table 1.27 Kamoa-Kakula 2020 PEA Capital Costs

Capital Costs (US$M)	Initial Capital (US$M)	Expansion Capital (US$M)	Sustaining Capital (US$M)	Total (US$M)
Underground Mining				
Underground Mining	156	995	5,620	6,772
Mining Infrastructure and Mobile Equipment	68	299	2,251	2,618
Capitalised Pre-Production	78	–	–	78
Subtotal	302	1,295	7,872	9,468
Power and Smelter				
Smelter Total	–	635	368	1,003
Power Supply Off Site	36	–	–	36
Subtotal	36	635	368	1,039
Concentrator and Tailings				
Plant	124	646	345	1,115
Tailings	15	26	550	591
Subtotal	139	672	895	1,706
Infrastructure				
Plant Infrastructure	69	536	678	1,283
Other Infrastructure	–	353	145	498
Overland Conveyors	–	118	66	183
Rail	–	72	84	156
Subtotal	69	1079	973	2,120
Indirects				
EPCM	37	111	35	184
Owners Cost	70	63	–	132
Customs Duties	9	137	361	507
Closure	–	–	308	308
Subtotal	116	311	704	1,130
Capital Expenditure Before Contingency	661	3,991	10,811	15,464
Contingency	53	470	1,147	1,670
Capital Expenditure After Contingency	715	4,461	11,958	17,134

Totals have been rounded.

Figure 1.23 compares the reported copper production in 2025 for the 20 highest producers by paid copper production. The Kamoa-Kakula 2020 PEA production is from the projected peak copper production which occurs in Year-8. Figure 1.24 shows the top 10 largest new greenfield copper projects defined as the 10 largest greenfield copper projects classified by Wood Mackenzie as “base case” or “probable” and ranked by nominal copper production (with the Kamoa-Kakula 2020 PEA and Kakula 2020 FS respective first 10 years’ average annual production of copper in concentrate considered to be its nominal copper production). The estimates are based on public disclosure and information gathered by Wood Mackenzie.

The Kamoa-Kakula IDP20 was not reviewed by Wood Mackenzie prior to filing.

Figure 1.23 2025 Predicted World Copper Producer Production

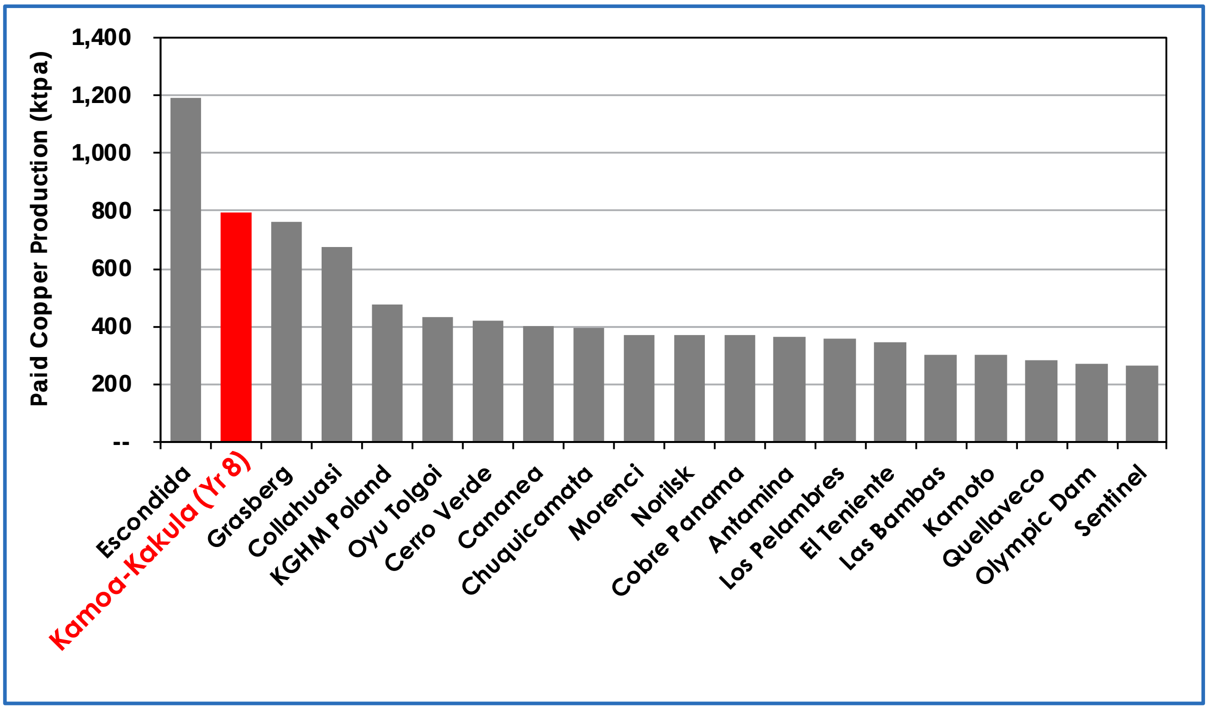


Figure by Ivanhoe, 2020. Source: Wood Mackenzie.

Figure 1.24 World Copper Producer Production and Head Grade

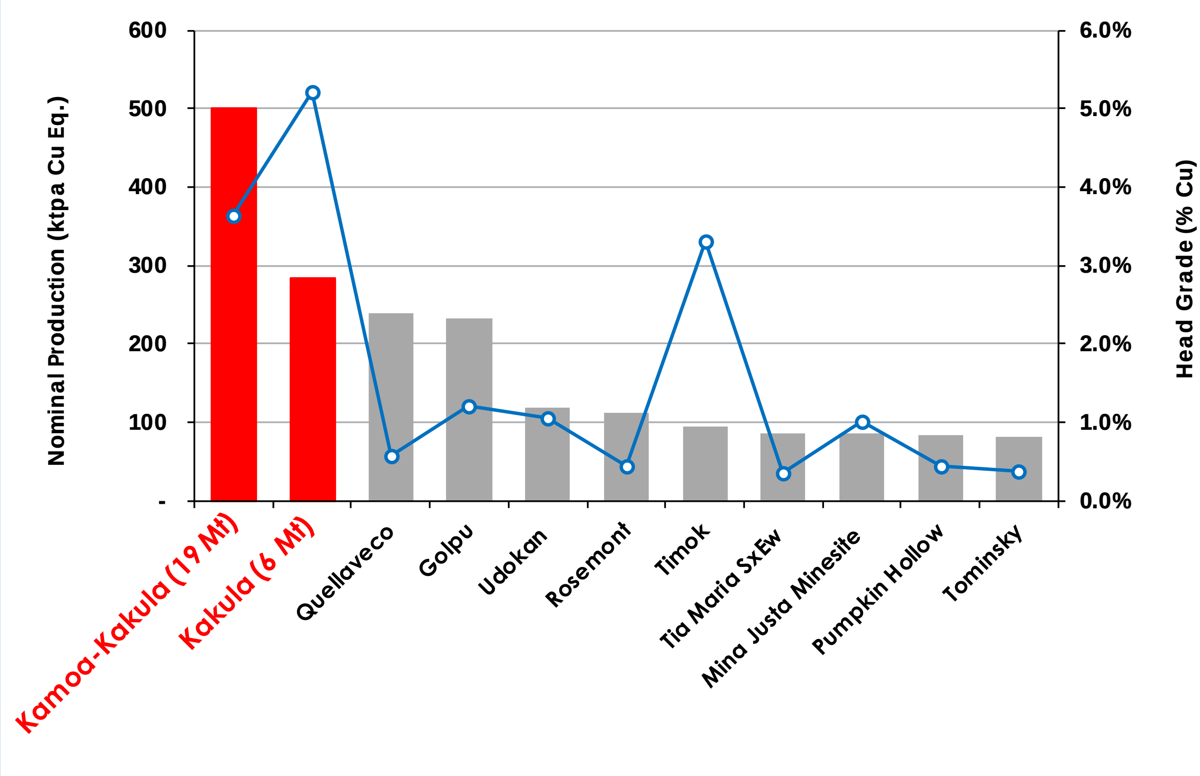


Figure by Ivanhoe, 2020. Source: Wood Mackenzie.

The after-tax net present value (NPV) sensitivity to metal price variation is shown in Table 1.28 for copper prices from US$2.00–US$4.50/lb.

The annual and cumulative cash flows are shown in Figure 1.25 (annual cash flow is shown on the left vertical axis and cumulative cash flow on the right axis).

Table 1.28 Kamoa-Kakula 2020 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**	**4.50**
Undiscounted	8,839	21,888	35,185	37,844	48,517	60,961	70,509
4.0%	4,620	11,251	18,056	19,416	24,876	31,243	36,089
6.0%	3,351	8,357	13,495	14,520	18,640	23,446	27,089
8.0%	2,422	6,323	10,320	11,117	14,318	18,054	20,876
10.0%	1,733	4,855	8,046	8,681	11,231	14,210	16,453
12.0%	1,213	3,771	6,374	6,891	8,968	11,393	13,214
15.0%	655	2,619	4,603	4,996	6,573	8,416	9,794
IRR (%)	21.1	37.8	53.2	56.2	67.3	79.9	89.0

Figure 1.25 Kamoa-Kakula 2020 PEA Projected Cumulative Cash Flow

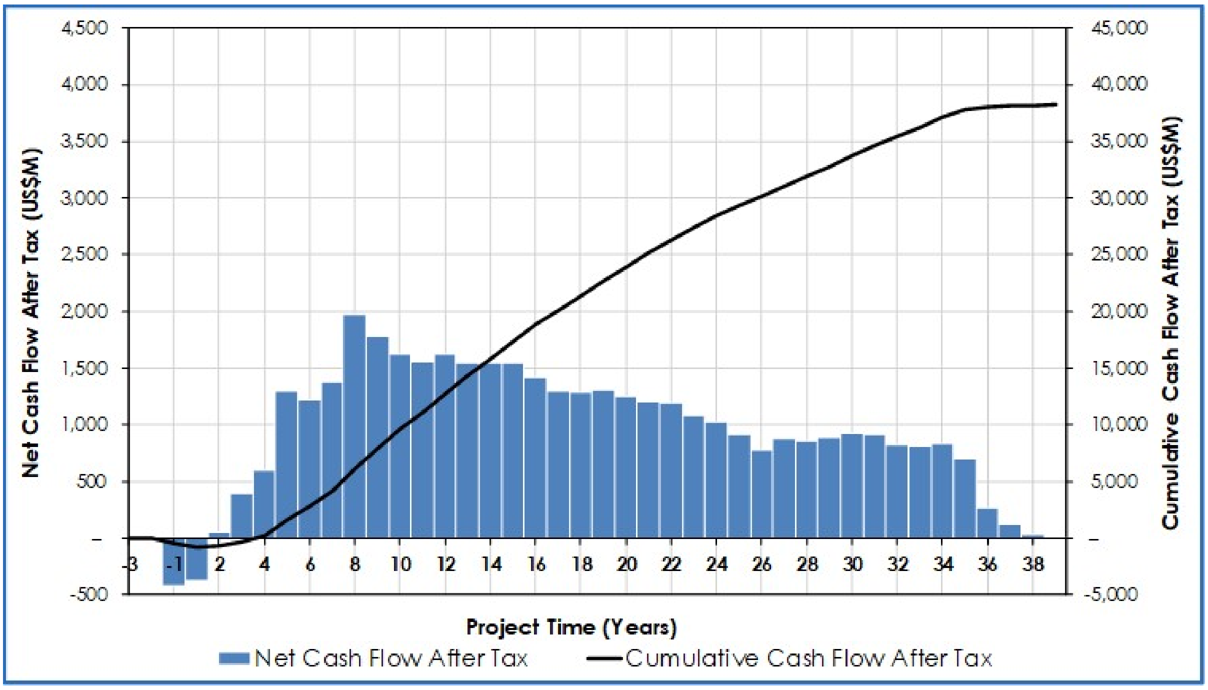


Figure by OreWin, 2020.

1.18.2 Kamoa-Kakula 2020 PEA Mining

Mining methods in the Kamoa-Kakula 2020 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 2020 FS, 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 2020 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.

1.18.3 Kamoa-Kakula 2020 PEA Processing and Infrastructure

The Kamoa-Kakula 2020 PEA scenario assumes that the project proceeds with first completing the Kakula 7.6 Mtpa process plant, built in two stages of 2 x 3.8 Mtpa, followed by a series of 3.8 Mtpa plant expansions to take the concentrator stream at the central complex's processing capacity to 19 Mtpa. The Kamoa-Kakula 2020 PEA has processing and infrastructure facilities that include:

- A 19 Mtpa combined processing facility complete with surface crushing and screening, milling, and flotation, consisting of five 3.8 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 1–5.

The Kakula process plant will be the first of five 3.8 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 7.6 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 products, 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 to be aligned with the mine production schedule. Phase 1 will treat 3.8 Mtpa in line with the mine ramp up and the throughput will be doubled during Phase 2 to 7.6 Mtpa. Refer to Section 24 for more detail on the Kakula design. A high-level block flow diagram of the Central Complex Concentrator 1 is presented in Figure 1.26.

Following the ramp-up of Central Complex Concentrator 1 to 7.6 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 6–15.2 Mtpa will also be completed in a two phased approach, as dictated by the mining plan, with a final 3.8 Mtpa expansion.

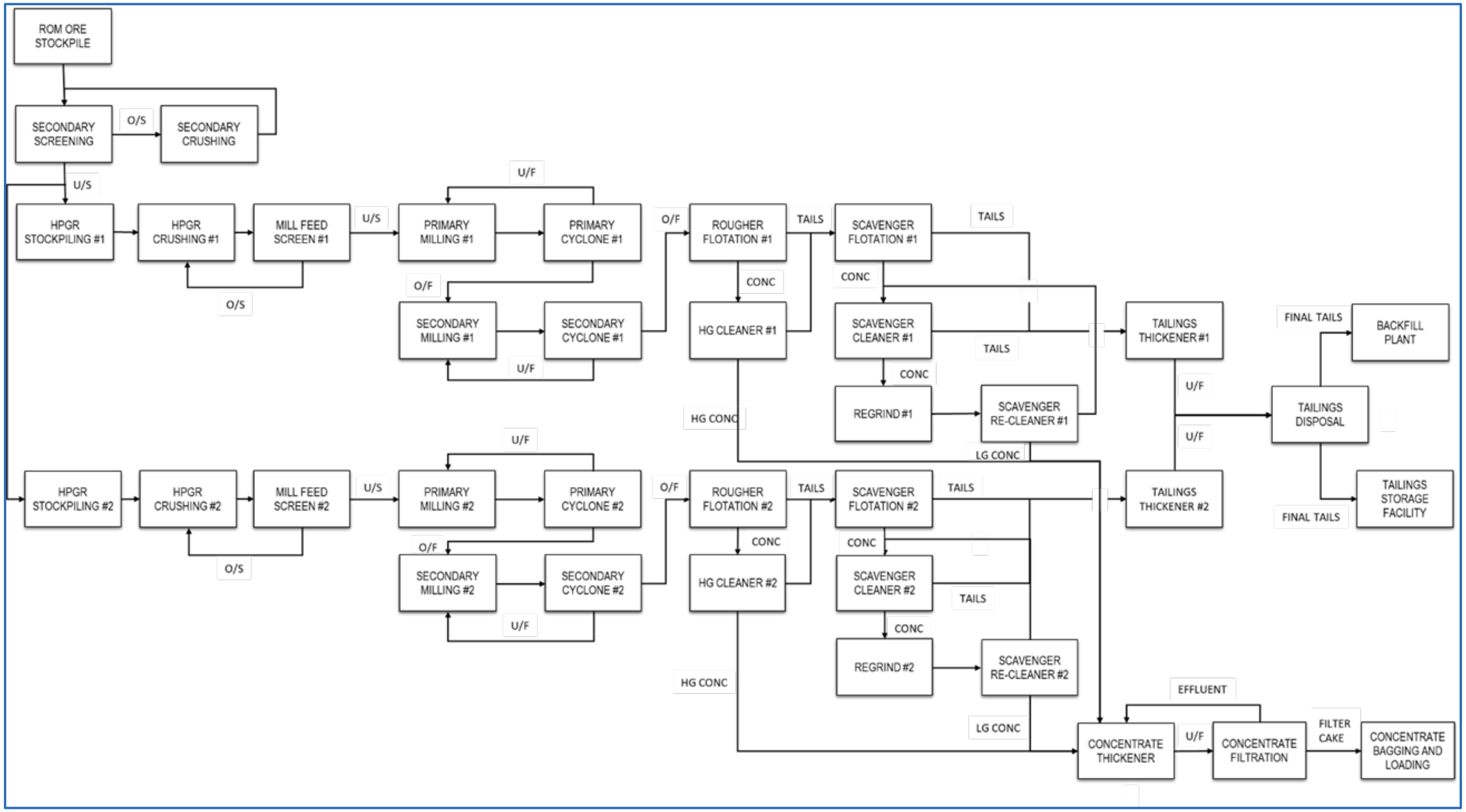
Concentrate will be conveyed from the adjacent concentrator complex into a concentrate shed located in the smelter complex.

The Kamoa-Kakula 2020 PEA also includes the construction of a smelter complex, based on Finnish-based Outotec's direct-to-blister furnace technology which is suitable for treating Kakula-type concentrates with relatively high copper/sulfur ratio, and low iron. China Nerin Engineering acted as the main engineering consultant with Outotec providing design and costing for propriety equipment, including the DBF furnace, waste heat boiler, and the slag cleaning electric furnace. The smelters design capacity is 1,000 ktpa of concentrate feed.

Concentrate is first dried in a steam dryer and sent to the DBF where it is smelted in the reaction shaft with oxygen-enriched air to produce molten slag containing oxide minerals, blister copper and sulfur-dioxide (SO_2) rich off gas. The oxidation reactions provide a portion of the heat required to melt the charge, with external fuel (in the form of pulverized coal and fuel oil) used to supplement the energy demand. Molten slag and blister copper collect in the DBF settler and are intermittently tapped (drained from the furnace) via dedicated tapholes. DBF slag, still containing appreciable amounts of copper, is further treated using metallurgical grade coke in an electric slag cleaning furnace (SCF) to recover oxidized copper in the form of blister. The SCF slag is slow cooled, crushed, milled, and processed by flotation to recover residual copper in the form of slag concentrate, which is recycled back to the DBF. The SO_2-rich off-gas is de-dusted, dried, and sent to a double-contact-double-adsorption acid plant for production of high strength sulfuric acid which is sold to the local market.

An on-site smelter offers numerous cost savings, including treatment charges, certain taxes and transportation costs. In addition, the sale of the sulfuric acid by-product would generate additional revenue. Sulfuric acid is in short supply in the DRC and is imported for use in processing ore from oxide copper deposits.

Figure 1.26 Central Complex Concentrator 1 Block Flow Diagram



1.19 Interpretation and Conclusions

1.19.1 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).

Wood has checked the data and estimation methodology used to construct the resource models (Datamine macros) and has validated the resource models. Wood 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.
  - At the Kakula deposit, the mineralisation appears more continuous compared to Kamoa.
  - In the Kakula south developments, minor offsets across growth faults have been encountered, but adjustments to the he mining methods has allowed the mining to follow the steeper dips of the mineralisation across the faults.
  - In the Kakula northern access drive, a larger growth fault was encountered where the mineralisation of the south side of the fault was faulted down (with variable offsets). A spiral decline was developed to accommodate the offsets, and re-established mining on the mineralisation.
  - The Kakula southern and northern declines and associated development are expected to join towards the end of 2020. This will provide a complete section across the deposit to further study the structural geology of the Kakula deposit.
- 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.

1.19.2 Kamoa-Kakula IDP20

The Kamoa-Kakula IDP20 includes an update of the Kamoa-Kakula (Kakula and Kansoko) Mineral Reserve and updates of the preliminary economic assessment (PEA) including analysis of the Kakula West and Kamoa North Mineral Resource.

The Kakula 2020 FS has identified a Mineral Reserve and development path that has confirmed significant value in the Kakula Deposit.

The Kakula-Kansoko 2020 PFS indicates the combined Kakula and Kansoko mine plans using the same 7.6 Mtpa processing facility of the Kakula 2020 FS generates significant value.

The analysis in the Kamoa-Kakula 2020 PEA indicates that the potential development scenarios for the Kamoa-Kakula project could include expansions and on site smelting that could provide potential additional project values. Considerable ongoing and additional studies need to be undertaken to define the development sequence and production rates including mining methods, plant sizing and location for the deposits, and identify the project potential.

1.20 Recommendations

1.20.1 Further Assessment

The 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 2020 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 Kansoko. 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 2020 FS has identified a Mineral Reserve and development path for the Kakula Deposit. It is recommended that studies at Kakula continue and incorporate a five year mining plan. This will include optimisation of the mine plan and monitoring of actuals against the budgets and design needs to be undertaken as the mine moves from development to production.
- The Kakula-Kansoko 2020 PFS has identified a Mineral Reserve. It is recommended that studies at Kansoko be progressed to feasibility study. The Kansoko study work needs to be prepared for feasibility and execution. This includes detailed plans and development of systems and procedures for of the controlled convergence room-and-pillar mining method.
- The Kamoa-Kakula 2020 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.

The three stages of the project provide a development plan. As development continues each stage of the project should be analysed and redefined.

1.20.2 Drill Programme

Extensive drilling has been completed at Kamoa and Kakula, and the goal of establishing sufficient Indicated Mineral Resources to support stand-alone mining operations at Kakula, Kakula West, Kansoko and Kamoa North has been achieved. The future drill plan at Kakula is to continue infill drilling in support of the current mine development, and to define the edges of the higher-grade material. 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. Wood has recommended further drilling of 17 km at a cost of $2.0M.

1.20.3 Processing Plant

Extensive metallurgical testwork have been conducted on the Kakula deposit which includes variability testing, locked cycle testing as well as mini pilot plant runs.

It is recommended to conduct further variability testing on the Kamoa deposit to confirm the suitability of the IFS4a flow sheet, and to provide more information regarding the variability of the Kamoa mineralogy and the associated impact on concentrate grade and recovery.

It is recommended to conduct more extensive testwork on the Kakula West and Kamoa North deposits to gain a better understanding of the grade-recovery profile when processed using the IFS4a and Kakula flow sheets.

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: the Kamoa-Kakula Project (the Project); the Platreef Project; and the Kipushi Project. Ivanhoe also holds interests in prospective mineral properties in the DRC and South Africa. These include an extensive, prospective a land package of ~3,000 km^2 in the Central African Copper belt adjoining the Kamoa-Kakula Project, known as the Western Foreland.

The Kamoa copper deposit discovery was made by Ivanplats Limited. Ivanplats Limited changed its name to Ivanhoe Mines Limited 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 39.6% interest in the Kamoa-Kakula project through 49.5% interest in Kamoa Holding Limited (Kamoa Holding).

2.2 Terms of Reference

The Kamoa-Kakula IDP20 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 following companies have undertaken work in preparation of Kamoa-Kakula IDP20:

- OreWin Pty Ltd.: Overall Report preparation, Kakula 2020 PEA analyses, Kakula and Kamoa North underground mining, Kamoa-Kakula combined production schedules, and financial models.
- Wood PLC: Geology, drillhole data validation, and Mineral Resource estimation for Kamoa and Kakula.
- SRK Consulting South Africa (Pty) Ltd.: FS and PFS Mine geotechnical recommendations.
- Stantec: Kakula 2020 FS and Kansoko 2020 PFS underground mine design.
- Golder Associates Pty Ltd: 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. China Nerin Engineering Co. Ltd: Smelter.
- KGHM Cuprum R&D Centre Ltd: Mine design, review the controlled convergence room-and-pillar mining method.
- Paterson and Cooke: Processing testwork.
- Outotec Oyj: Processing testwork.

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 1.6, 1.16, 1.16.3, 1.16.4, 1.17, 1.17.3, 1.17.4, 1.18, 1.18.1, 1.18.2, 1.19, 1.20, 1.20.1; Section 2; Section 3; Section 4; Section 5; Section 6; Section 19; Section 20; Sections 21.1, 21.2, 21.2.3, 21.3, 21.3.3, 21.4; Section 22; Section 23; Sections 24.1 to 24.5, 24.7; Section 25.2; Section 26.1; Section 27.
- Gordon Seibel, SME Registered Member (2894840), Principal Geologist, Wood plc, was responsible for: Sections 1.7 to 1.11, 1.13, 1.14, 1.19.1, 1.20.1, 1.20.2; Section 2.2 to 2.6; Section 3; Section 7; Section 8; Section 9; Sections 10.1 to 10.5, 10.8, 10.9; Section 11.1 to 11.2, 11.4 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 Corporate Consultant, was responsible for: Section 2; Section 10.6; Section 16.1; and Section 16.2; Section 27.
- Jon Treen P. Eng (Mining), PEO (90402637), employed by Stantec Consulting International LLC as Mining Business Line Leader, was responsible for: Sections 1.15, 1.16.1, 1.17.1, 1.19.2, 1.20.1; Section 2; Section 15; Section 16.3, 16.4; Sections 21.1, 21.2.1, 21.3.1, 21.4; Section 25.3; Section 26.3; Section 27.
- Marius Phillips, MAusIMM (CP 227570), Vice President Process, DRA Global, was responsible for: Sections 1.12, 1.16.2, 1.17.2, 1.18.3, 1.19, 1.20, 1.20.1, 1.20.3; Section 2; Section 10.7; Section 11.3; Section 13; Section 17; Section 24.6; Section 25.4; Section 26.4; Section 27.
- Alwyn Scholz, B.Eng., MSc, Pr.Eng, ECSA (20150110) employed by DRA Global as Study Manager, was responsible for: Section 1.19, 1.20, 1.20.1, Section 2; Section 18; Sections 21.1, 21.2.2, 21.3.2, 21.4; 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 15–17 February 2010, from 27–30 April 2010, on 15 November 2012, from 12–14 September 2015, from 24–25 October 2016, on 28–29 June 2017 and from 6–8 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.

Mr. Gordon Seibel visited the Project from 9–10 February 2011, from 5–8 November 2011, from 12–14 November 2012, from 18–22 January 2016, and from 7–8 February 2020. 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 February 2020 visit, Mr. Seibel checked drillhole locations at Kamoa North, inspected drill core from the Bonanza Zone, and included a visit underground at Kakula to view the exposed portions of the Kakula orebody.

Mr. William Joughin visited the site from 10–13 July 2017 and 13–16 August 2018 to review the geotechnical core logging and to inspect the ground conditions and support in the Kansoko decline and Kakula box-cut and decline 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–1 November 2013 and the Kamoa and Kakula sites from 6–8 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 raise. 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 26–28 January 2016, involved reviews of Lubin Mine, Runda Mine, and a review with their technology group Cuprum.

Mr Marius Phillips from DRA Global have not visited the site due to travel restrictions caused by COVID19, 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 19–23 August 2019 and 12–14 November 2019. The site visit by Mr. de Beer included briefings from Ivanhoe personnel, site inspections of the Kakula decline portal and box cut, potential areas for mining, monthly construction progress meetings, plant and infrastructure, and a review of the existing infrastructure and facilities in the local area around the Kakula site.
- Mr. Danie Oosthuizen, Senior Civil Project Engineer for Kamoa-Kakula project employed by DRA Global, has visited the Kakula site on the following occasions: 11–18 September 2019, 15–22 January 2020 and 9–13 February 2020. The site visit by Mr. Oosthuizen included site inspections of the existing infrastructure, new infrastructure currently being constructed 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.
- Mr. Alwyn Scholtz, Study Manager for Kamoa-Kakula project employed by DRA Global, has visited the Kakula site from 18–23 August 2019 and between 6–7 August 2018. The site visit by Mr. Scholtz included site inspections of the Kakula declines, surface workshops, raise boring sites, offices, camps, roads, surface water settling dams, and other surface buildings.

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

Table 2.1 SRK Site Visits

Person	Dates	Purpose
Jarek Jakubec	27 April–1 May 2010	Initial project geotechnical review.
Wayne Barnett	21–25 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	22–27 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	5–12 August 2011	Geotechnical logging QA/QC.
Wayne Barnett	12–17 August 2011	Review the structural geology model development; review and update based on new drill core and orientated core measurements.
Ross Greenwood	12–19 February 2012	Geotechnical data collection QA/QC.
Wayne Barnett	13–17 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	18–20 November 2014	Geotechnical 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–01 August 2016	Quality control. Decline Ground Support Recommendations.
Rory Bush	11–21 November 2016	Quality control. Geotechnical logging for the Kakula Decline Ground Support Recommendations.
William Joughin and Denisha Sewnun	10–13 July 2017	Geotechnical Review of the Kansoko Decline and Kakula box-cut Ground Support. Recommendations. Quality control.
William Joughin, Joseph Muaka and Philani Mpunzi	13–16 August 2018	The ground conditions and construction of the box-cut inspections underground conditions review, QA/QC on core logging.
Shaun Murphy and Denisha Sewnun	11–15 November 2019	Quality Control on geotechnical logging, assessment of the underground ground conditions at the mine site, rock sampling.

2.5 Effective Date

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

- Effective date of the Report: 13 October 2020.
- Date of the database closure Kamoa Mineral Resource estimate: 20 January 2020.
- Date of the database closure Kakula Mineral Resource estimate: 1 November 2018.
- Date of Mineral Resource estimate for Kamoa: 30 January 2020.
- Date of Mineral Resource estimate for Kakula: 10 February 2020.
- The Mineral Resources at Kakula were estimated as of 10 November 2018. On 10 February 2020, the inputs used in assessing reasonable prospects of eventual extraction and the drill data inputs were reviewed to ensure the estimate remained current. There are no changes to the estimate as a result of the review, and the estimate has an effective date of 10 February 2020.
- Date of the Mineral Reserve estimate for Kamoa: 8 September 2020.
- Date of the Mineral Reserve estimate for Kakula: 8 September 2020.
- 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 IDP20, 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: Kakula 2020 FS Section 19 Property Description and Location, October 2020.
- 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 14.3 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: Kakula 2020 FS, Section 19 Property Description and Location, October 2020.

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 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: Kakula 2020 FS, Section 19 Property Description and Location, October 2020.

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: Kakula 2020 FS, Section 19 Property Description and Location, October 2020.
- Kamoa Copper SA: Kakula 2020 FS, Section 18 Environmental and Social, October 2020.
- 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 Constraints 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:

- Tedrow Consulting, 2020: Letter from David Guarnieri and Thomas Jolivet to David Van Heerden regarding content of the bankable feasibility study required for the calculation of the Super-profits tax, 10 August 2020.
- Ivanhoe Mines Ltd.: Email Re: Updated Taxes and Royalties to OreWin, 3 October 2020.
- Kamoa Copper SA: Kakula 2020 FS, Section 19 Property Description and Location, October 2020.

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. It is located approximately 25 km west of the town of Kolwezi, and about 270 km west of the regional centre of Lubumbashi.

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

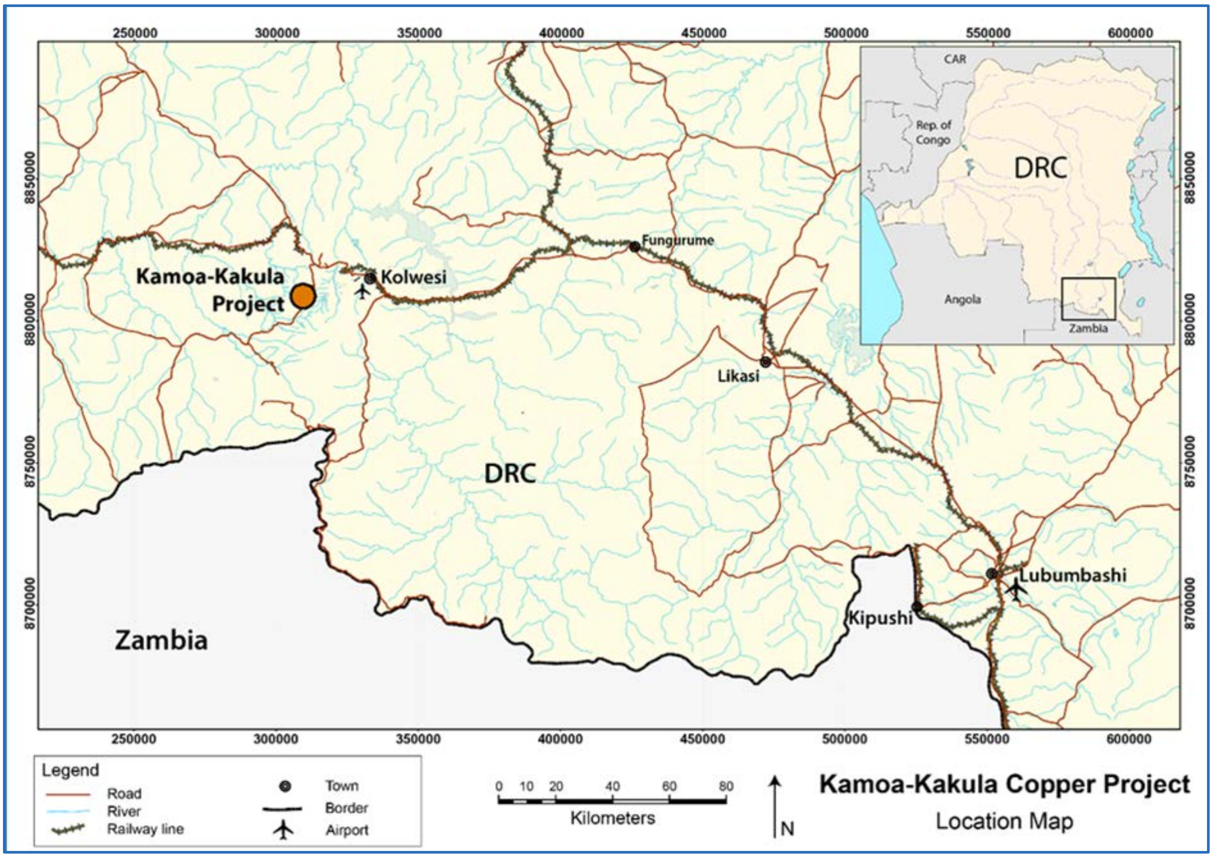


Figure by Ivanhoe, 2016.

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 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 SA, a share transfer agreement that transferred an additional 15% interest in the Project to 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 Project, while Crystal River Global Limited holds an indirect 0.8% interest and the DRC holds a direct 20% interest in the 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 SA's share capital – to the DRC, in consideration for a nominal cash payment and other guarantees from the DRC summarised below. In addition, the DRC owns 100 non-dilutable Class B shares, representing 5% of Kamoa Copper SA'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 Project.
- Kamoa Holding and the DRC 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 approximately US$1.24 billion on 31 December 2019), and 100% of any financing of the project by third parties.
- The DRC confirmed that the Project will be developed with the support of the government of DRC and of its Ministry of Mines by Kamoa Copper SA with the current and future shareholders of Kamoa Holding.
- The DRC acknowledged and confirmed that all permits and mining rights currently held by Kamoa Copper SA in respect of the 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 SA'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 SA.
- The DRC confirmed and guaranteed that the Project will not be subject to any taxes or duties other than those legally required by the applicable statutory and regulatory provisions.
- The DRC 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 SA's request and subject to the satisfaction of the applicable conditions, the DRC State shall provide its assistance to Kamoa Copper SA, its affiliates and subcontractors for the purpose of obtaining the advantages contemplated by the DRC'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 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 SA, 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 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 former Katanga 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 (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, now called Gécamines SA (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 (DPEM).

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

- The Prime Minister, who is notably responsible for enacting the Mining Regulations for the implementation of the Mining Code and declaring mineral substances as being a strategic mineral substance.
- The Prime Minister exercises his rights by decrees, adopted in Council of Ministers, upon proposal of the Minister of Mines and, where appropriate, the relevant Ministers.
- The Minister of Mines, who has notably jurisdiction over the granting, refusal and withdrawal of mining rights.
- The 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 Mines is notably responsible for controlling and monitoring the performance of activities in relation to mines in accordance with legal and regulatory provisions in force.
- The DPEM is notably responsible, in collaboration with the Congolese Agency for Environment, the national fund of promotion and social service and, where appropriate, any other relevant body of the State, for implementing the mining regulations concerning environment protection and performing the environmental examination of environmental and social impact studies and environmental and social management plans. These administrations are also notably responsible for controlling and monitoring, without limitation, the obligations of the holders of mining rights concerning health and safety and the protection of environment in the sector of mines.
- 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 Procurement

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

Law No.18/001 dated 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 potential financial consequences for Kamoa Copper SA.

- Kamoa Copper SA and Kamoa Holding, as well as the owners 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 remain entitled to the 10-year stability guarantee granted by Article 276 of the 2002 Mining Code.
- However, since Law No.18/001 came into force, the DRC has applied the more stringent tax requirements adopted therein to all mining companies, including Kamoa Copper SA. Kamoa Copper SA, Kamoa Holding and the owners of the shares in Kamoa Holding contest this approach and rely on, inter alia, the DRC's contractual commitments and the stability guarantee.
- In the interim, until such time as this issue is finally resolved, Kamoa Copper SA pays the taxes imposed by the DRC, under duress. This is done solely for the purpose of preventing, as far as possible, the damages that could result from sanctions imposed on Kamoa Copper SA, while clarifying that such payments cannot be considered as a waiver to any of the rights of Kamoa Copper SA and in particular the stability guarantee.
- Increased tax and customs requirements, reinforced by the breach by the deviation from the stability guarantee.

  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.

Also see the comments in Section 4.2.8 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. Kamoa Copper SA is in the process of finalising the cahier des charges with a target to have it approved, subject to potential delays that could notably result from the consequences of the global COVID-19 outbreak.
- 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 SA 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 (Subcontracting Law).

- These new rules increase the costs of the Project and could be considered as being contradictory, without limitation, to the stability guarantee to which Kamoa Copper SA is entitled. It is also incongruent with Article 273 f of the Mining Code, which provides 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. This priority must be given for all contracts in relation to the mining project, provided that the Congolese business offer equivalent conditions in terms of quantity, quality, price, delivery deadlines and payment.
- Kamoa Copper SA is nevertheless doing its best efforts to ensure voluntary compliance with the new requirements, without waiving its rights under the stability guarantee and the share transfer agreement. This includes ongoing and increasing development of local suppliers and voluntary compliance with laws regarding main contractors and subcontractors.
- Kamoa Copper SA also monitors the regulatory provisions adopted or to be adopted to ensure, as far as possible, adequate enforcement of the relevant legislative requirements.
- Kamoa Copper SA however notes that the new authority governing subcontracting in the private sector (ARSP) requires all subcontractors to be approved by ARSP and to pay 5% of their contract to ARSP. There are nevertheless pending issues concerning who are the subcontractors and subcontracting agreements concerned by these expectations from ARSP, the legality of which is currently challenged by several businesses in DRC.
- There is a pending engagement, initiated in February 2020 by the Government of DRC that should result shortly into new regulations aiming at clarifying the implementation of the Subcontracting Law.
- 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).
- The obligation to transfer an additional 5% of the shares in the share capital of the company upon each renewal of the exploitation permits.

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

- The minerals substances declared as being strategic substances that can be changed anytime by a decree from the Prime Minister deliberated in Council of Ministers, upon an opinion from the relevant sectorial Ministers, the royalty applicable to such strategic substances being 10%.
- Pursuant to Decree No.18/042 dated 24 November 2018, cobalt, germanium and colombo-tantalite (coltan) were declared as being “strategic mineral substances”.
- The mining products for sale that must be compliant with the nomenclature set out by the applicable regulations that could possibly change, from time to time, by DRC Government.

- On 16 February 2019, the Ministry of Mines and the Ministry of Finances in particular signed an interministerial order amending Article 5 of the interministerial order dated 21 November 2018 setting out the nomenclature of mining products for sale (produits miniers marchands) (hereinafter referred to as the "Interministerial Order") as follows:
  - The export of concentrates of copper is prohibited.
  - The provisions of points 1 and 2 of point VI relating to simple concentrate and of point 1 of point VII relating to mixt concentrate of the appendix to the Interministerial Order only apply for local sale.
- However, on 20 March 2019, the Minister of Mines signed a letter, copying the Minister of Finances, explaining that after consultation with the Ministry of Finances, a moratorium going up to the resolution of the energy deficit issue was granted to mining companies producing notably concentrates of copper and clarifying that an assessment would be performed every semester by the Mine Department together with mining companies to consider the opportunity of lifting or maintaining this moratorium. The Minister of Mines nevertheless highlighted the absolute necessity for mining companies producing concentrates of copper to do what is necessary to obtain, from their treatment process, more elaborated mining products for sale in order to enable them, as well as the State, to obtain better returns on mining production.
- In spite of the absence of amendment of the Interministerial Order, the export ban was lifted in practice on the basis of the Minister of Mines dated 20 March 2019 and companies continued exporting copper concentrates.
- By a letter dated 22 February 2020, the new Minister of Mines informed mining companies that the copper concentrate export ban will resume as from 22 August 2020 considering the existing capacity of transformation of the production of copper concentrates within DRC.
- It seems that DRC considered for this purpose the current production and not the production to come (such as the one from Kamoa Copper SA).
- There are technical issues for the Project related to this Government's objective and associated concerns for Kamoa Copper SA that are currently examined and will be addressed through engagement with DRC in a timely manner.

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

Kamoa Copper SA, Kamoa Holding and the owners of the shares of Kamoa Holding consider that Kamoa Copper SA 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 SA is entitled to in accordance with Article 276 of the 2002 Mining Code and the share transfer agreement entered into between the DRC and Kamoa Holding. They nevertheless note the current contrary interpretation adopted by the DRC government.

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 exceed 15 years until the end of the mine's life, if conditions laid out in the Mining Code are met.

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

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

11. Enter within the exploitation perimeter to proceed with mining operations.
12. Build the facilities and infrastructure required for mining exploitation.
13. 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.
14. Dispose (disposer), transport and freely market the products for sale originating from within the exploitation perimeter.
15. 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.
16. 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.
- Undertaken to comply with the cahier des charges defining the social responsibility vis-à-vis 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 octies of the Mining Code, the trading of the mining products that originate from exploitation perimeters must be done in accordance with the laws and regulations in force in DRC. This provision also specifies that the holder of an exploitation permit may sell its products to clients of its choice at fair price with regard to market conditions.

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

Under the Mining Code, a mining rights holder must pay in a timely manner a levy on the total surface area of his mining title (Article 238 of the Mining Code). Levies are defined on a per hectare basis, and increase on a sliding scale for each year that the mining right is held, until the third year, after which the rate remains constant. In this Report, this levy is referred to as a “tax on the area of mining 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 squares held by a title holder. Different levels of duties are levied depending on the number of years a mining title is held, and whether the mining right is an exploration or exploitation mining right. In this Report, this tax is referred to as annual superficiary rights”.

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, a 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 SA was granted with such an authorisation from the Governor of the Province on 23 July 2014. Kamoa Copper SA nevertheless noted a typing error in one of the mining rights referred to in the above mentioned authorisation and subsequently to a meeting in this respect with the Provincial

Minister of Mines for Lualaba, Kamoa Copper SA is in the process of preparing a letter to the relevant authorities to confirm, as soon as practically 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. This guarantee can be provided by means of a bank guarantee. Funds posted as guarantee are not at the disposal of the DPEM and are to be used for the rehabilitation of a mining site. Kamoa Copper SA complied with its obligation in this respect, in accordance with the instalments set out in the approved updated environmental impact study of the Project. However, when DPEM examined Kamoa Copper SA's new draft updated ESIS filed on 23 December 2019, it raised certain queries regarding the validity of the financial guarantees previously filed for the 4 last instalments.

Kamoa Copper SA is in the process of addressing the queries of the DPEM in this respect in order to have its new draft updated ESIS promptly approved.

With regard to DRC's expectations and in order to mitigate risks, Kamoa Copper SA also applied for an authorization covering all its currently operated classified facilities, as well as its classified facilities planned in the near future, as well as for clearing authorisations (permis de déboisement) and paid, under duress, the taxes required by the relevant administrations, while they were, in Kamoa Copper SA's view, legally challengeable. Kamoa Copper SA is actively following those applications with the relevant administrations to ensure it is promptly granted with the authorisations applied for and paid for or to be paid for, when required by the relevant administrations.

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 ESIS.
- When a control and/or monitoring report demonstrates that the mitigation and rehabilitation measures planned in its PEMP are no longer adapted and that there is a significant risk of adverse impact for the environment.

The Mining 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 2-year environmental audit concerning Exploitation Permits No. 12873, 13025 and 13026 was thus filed on 07 June 2018 and Kamoa Copper SA is in the process of preparing the new 2-year environmental audit.

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

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

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.

At the date of this Report, copper products that Kamoa Copper SA intends to sale and export are not listed among the strategic mineral substances.

Pursuant notably to Article 276 of the 2002 Mining Code and insofar as Kamoa Copper SA holds mining rights that were valid when Law No.18/001 entered into force, Kamoa Copper SA, Kamoa Holding and the owner of the shares of Kamoa Holding consider that Kamoa Copper SA 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 SA nevertheless notes the contrary interpretation from DRC and its administrations on similar issues and the opinion from Kamoa Copper SA, Kamoa Holding and the owners of the shares in Kamoa Holding is that in the event DRC would impose Kamoa Copper SA 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 SA 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) . 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.

Table 4.1 Permit Summary Table

Exploitation Permit (PE) No.	Grant Date	Expiry Date	Mineral/Metal Rights Granted	Number of Cadastral Squares	Area (ha)*
12873	20 Aug 2012	19 Aug 2042	Silver, Bismuth, Cadmium, Cobalt, Copper, Iron, Germanium, Nickel, Gold, Palladium, Platinum, Lead, Rhenium, Sulfur and Zinc.	62	5,207.67
13025	20 Aug 2012	19 Aug 2042	Silver, Bismuth, Cadmium, Cobalt, Copper, Iron, Germanium, Nickel, Gold, Palladium, Platinum, Lead, Rhenium, Sulfur and Zinc.	204	17,135.69
13026	20 Aug 2012	19 Aug 2042	Silver, Bismuth, Cadmium, Cobalt, Copper, Iron, Germanium, Nickel, Gold, Palladium, Platinum, Lead, Rhenium, Sulfur and Zinc.	202	16,972.25
Sub Total					39,315.61

*The above-mentioned areas are approximate and subject to GIS verification.

Figure 4.2 Project Tenure Plan

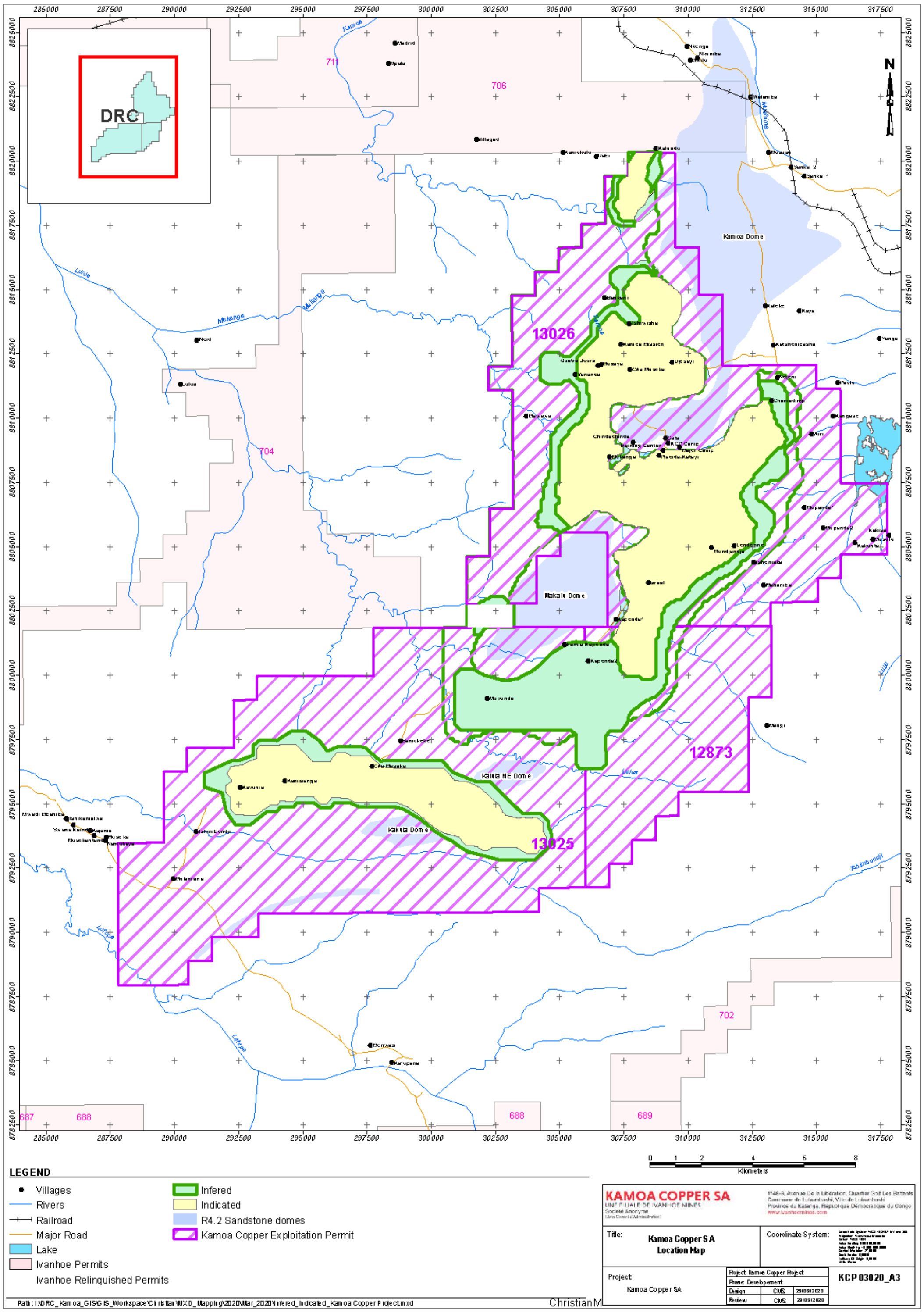


Figure by Kamoa Copper SA, 2020.

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 31 January of each year. Ivanhoe advised the QPs that the required payments for 2019 and 2020 were made for the three above-mentioned Exploitation Permits.

Ivanhoe is also actively exploring in other areas of the DRC close to the perimeters of the mining rights constituting the Project.

4.4 Surface Rights

At the effective date of this Report, Kamoa Copper SA holds no surface rights in the Project area. However, subject to the comments set out in Section 4.2.6, Kamoa Copper SA 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 footprint inside 48 km^2 enclosing 129 households surveyed out of whom 45 have been physically relocate after complete field compensation and land replacement. The field compensation, land replacement and physical relocation are in progress for the rest of households. 16 km of the mine area was fenced off.

Kamoa Copper SA also planned 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 SA 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 Property Agreements

There are no property 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 a new gravel road built directly from Kakula that joins the main Kolwezi-Lubumbashi tarred road at the Kolwezi airport, just south of the city. On site, sealed gravel roads have been built between the Kamoa Camp, Kansoko Mine and Kakula Mine 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 developing infrastructure required to support mining and processing operations. More than 23 kilometres of underground development has been completed at the Kakula Mine, and the Project has issued purchase orders for the long-lead mining and processing equipment. The first oversized loads of equipment for the processing plant arrived at the mine on February 21, 2020. Initial copper concentrate production from the Kakula Mine is scheduled for the third quarter of 2021.

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 modernisation project is expected to restore Mwadingusha to its installed capacity of about 71 MW. After completion and hand over, HPP Mwadingusha 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 1971–1975, the Tenke Fungurume Consortium (consisting of Amoco, Charter, Mitsui, BRGM and L. Tempelsman, and operated as the Société 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 Wood plc) for the Kamoa deposit in 2009 (Parker H., 2009) and the estimate was updated in 2010, 2011, 2012, 2013, 2016, 2017, 2018, 2019, and has now been updated in 2020.

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 6.0 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 4.0 Mtpa to 6.0 Mtpa, and the total combined production rate has increased from 8.0 Mtpa to 12.0 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 Technical Report titled Kamoa-Kakula 2018 Resource Update with an effective date in March 2018 included a restatement of the Kamoa-Kakula 2017 Development Plan.

The Technical Report titled Kamoa-Kakula Integrated Development Plan 2019 with an effective date in March 2019 included: Mineral Reserves for Kamoa, initial Mineral Reserves for Kakula, Kakula and Kakula West Mineral Resource updates, and the Kamoa-Kakula 2019 PEA considering an 18 Mtpa plant expansion.

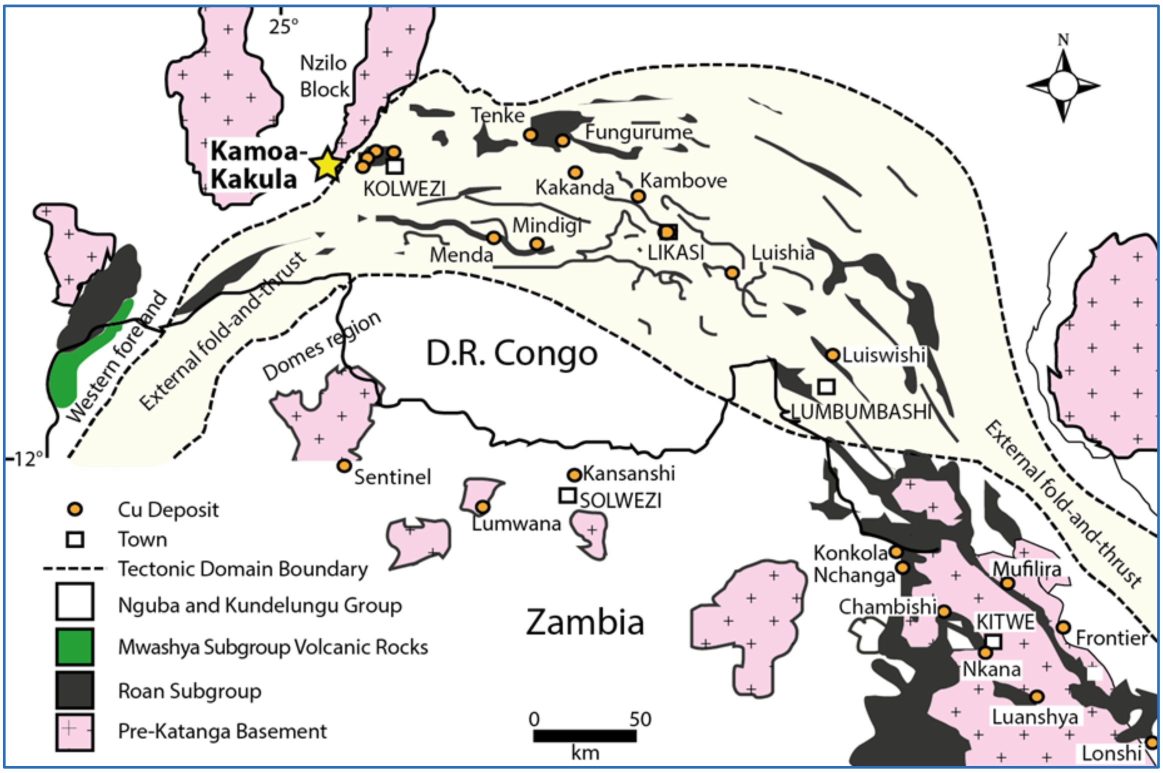
The previous Technical Report was the Kamoa-Kakula 2020 Resource Update with an effective date in March 2020. This included an update to the Kamoa Mineral Resource and a restatement of the Kamoa-Kakula Integrated Development Plan 2019.

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology

The metallogenic province of the Central African Copperbelt is hosted in metasedimentary rocks of the Neoproterozoic Katanga Basin, an evolving intracontinental rift. The Katangan Basin overlies a composite basement consisting of older, multiply-deformed and metamorphosed intrusions that are mostly of granitic affinity and supracrustal metavolcanic–sedimentary sequences. The lowermost, continental siliciclastic rock sequences within the Katangan Basin 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-Kakula 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). These rocks are overlain by thick diamictite (the base of which hosts the Kamoa-Kakula deposit), carbonate rocks and relatively monotonous, non-evaporitic siliciclastic rocks of the N’Guba and Kundulungu Groups. During this deposition, there was a progressive widening of the basin that resulted in younger strata being deposited onto the basement rocks at the basin periphery (Selley et al., 2018). Basin inversion occurred during the Lufilian Orogeny, with the shape of the orogen defined by a convex-northward array of folds and reverse faults (the Lufilian Arc), that are clearly shown by the curvilinear outcrop patterns of Roan Group strata in the Katangan portion of the Copperbelt (Figure 7.1).

Figure 7.1 Geological Setting Central African Copperbelt

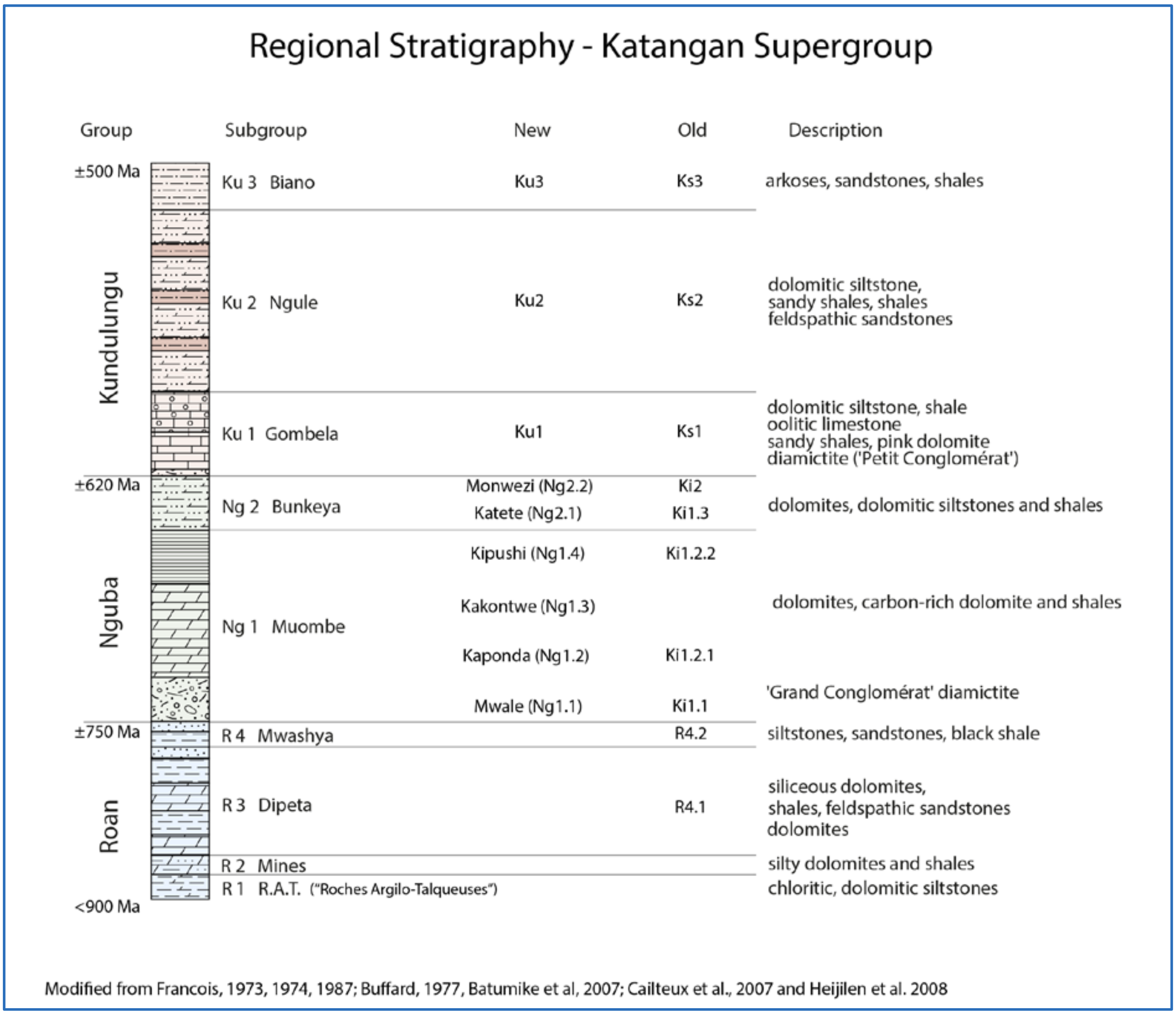


Source: Adapted from Schmandt et al (2013).

All of the Mines Subgroup copper (± cobalt) orebodies of the Katangan Copperbelt occur as mega fragments (écailles) up to kilometres in size, within a megabreccia. 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 Katangan Supergroup within the Katanga Basin in the DRC sector is currently subdivided into the Roan (R), N'Guba (Ng) and Kundulungu (Ku) Groups, (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 by Ivanhoe geologists, and thus in this Report, use the earlier nomenclature.

Figure 7.2 Stratigraphic Sequence, Katangan Copperbelt



7.2 Project Geology

The modelled Kamoa deposit is located in a broadly-folded terrane, with the antiform centred on the Kamoa and Makalu domes. The central portions of Kakula are located on the southern extension of this antiform, with Kakula West located on the top of a separate, but parallel trending antiform. The domes form erosional windows exposing the redox boundary between the underlying haematitic (oxidised) Roan sandstones (Mwashya Subgroup), and the overlying carbonaceous and sulfidic (reduced) Grand Conglomérat diamictite (N'Guba Group), which comprises diamictites with minor interbedded sandstone, siltstone, and conglomerate. The mineralisation at Kamoa-Kakula is hosted towards the base of the Grand Conglomérat unit (Ki1.1).

Although the term diamictite is often associated with glacial deposits, the diamictites of the Grand Conglomérat at Kamoa are interpreted as 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 only forms 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 deposition of the copper sulfide mineralisation in the diamictites and siltstone units at the base of the Grand Conglomérat (Schmandt et al. 2013).

Andesite/dolerite sills occur as one or more, 5–80 m thick, apparently concordant tabular bodies in the extreme northeast of the Project area. The Katangan rocks in the Project area are weakly metamorphosed to lower greenschist facies. Alteration minerals include carbonate, chlorite, sericite, potassium feldspar, and hematite.

Two primary structural 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 Fault zone of the central Congolese Copperbelt (Selley et al., 2018). Basin growth during deposition of the Grand Conglomérat is evident in a progressive thickening to the south-west.

Mineralisation at Kamoa-Kakula has been defined over an irregularly-shaped area of about 28 km x 23 km. Mineralisation is typically stratiform, and vertically zoned from the base upward with chalcocite (Cu_2S), bornite (Cu_5FeS_4) and chalcopyrite ($CuFeS_2$). The nature of the copper grade distribution is related to its stratigraphic position, proximity to the Roan aquifer (or structures that may have focussed fluid flow), and the localised development of lithological units. The earliest sulfide mineralisation at Kamoa-Kakula was deposited during diagenesis and formed abundant framboidal and cubic pyrite in the laminated siltstones (Schmandt et al, 2013). This pyrite mineralisation above the mineralised horizon could possibly be exploited to produce pyrite concentrates for sulfuric acid production (needed at oxide copper mines in the DRC).

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 7.3).

Figure 7.3 Prospect Areas Within the Combined Exploitation Permits

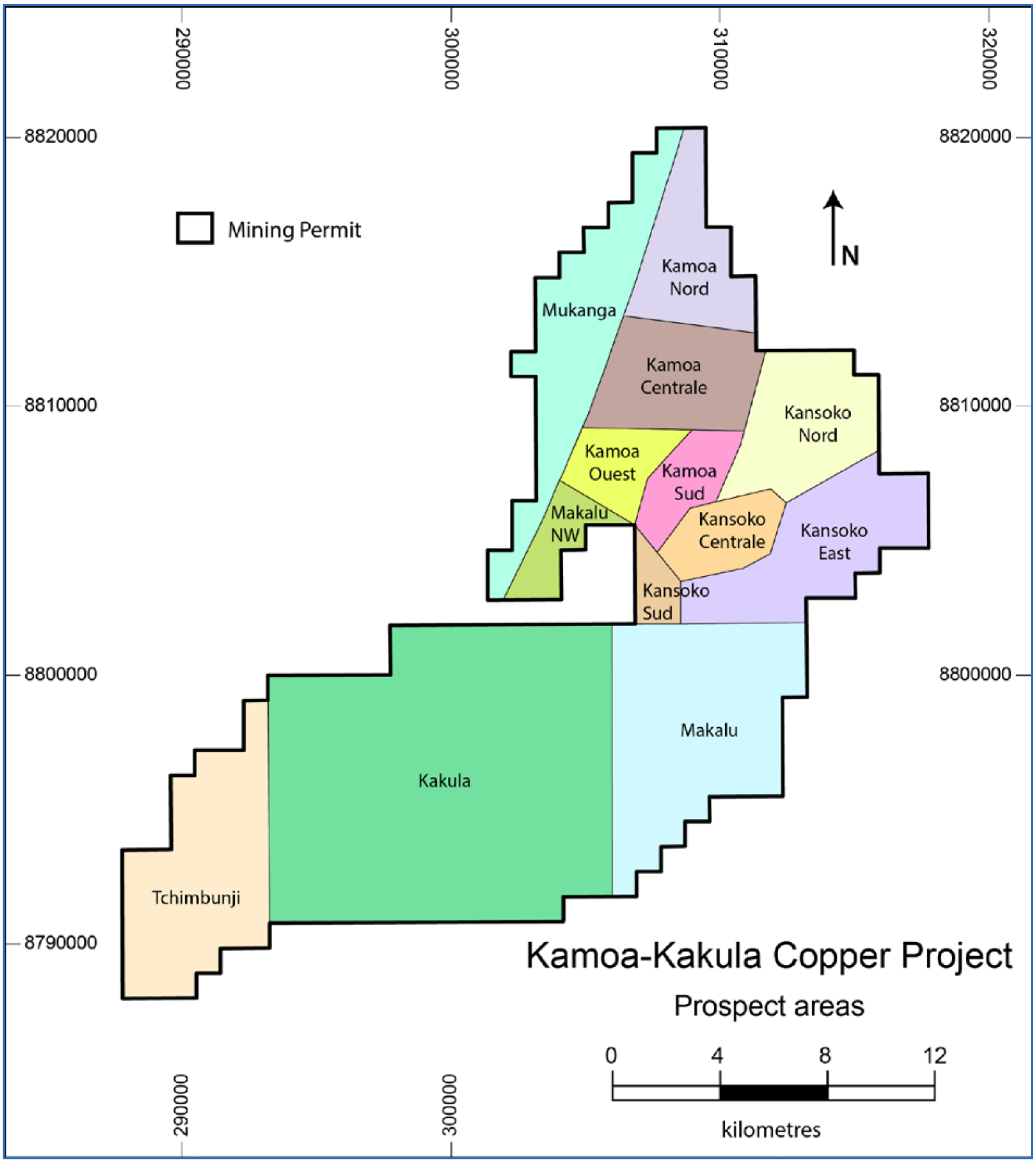


Figure provided by Ivanhoe, 2020.

7.3 Kamoa Deposit

7.3.1 Lithologies

At Kamoa, haematite-bearing sandstone and siltstone of the Mwashya Subgroup (upper Roan Group) (R4.2) form the oxidised lower strata. The pyritic rocks of the basal diamictite and inter-bedded siltstone-sandstones form the reduced host rock (Twite et al. 2018). Two units are recognised within the basal diamictite, a clast-rich diamictite (Ki1.1.1.1), which is overlain by a clast-poor diamictite (Ki1.1.1.3). Mineralisation is typically concentrated along the basal contact of this clast-poor diamictite, or in a locally-developed intermediate siltstone (Ki1.1.1.2) that separates the two diamictite units. 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. Sandy or gritty layers are developed within the siltstone, and conglomerate layers are locally developed towards the base of the unit. Pyrite can range from fine to coarse-grained. The basal contact of the KPS is marked by very finely layered varves. Dropstones 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 bedded to laminated pyritic siltstone with intercalations of sandstone and minor gritty pebbles. The Ki1.1.4 is overlain by a thick (>300 m) unit of 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.

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

7.3.2 Thickness of Stratigraphic Units

Vertical thickness trends in the different stratigraphic units indicate a variable orientations of the basin controlling structures that were active during sedimentation (Figure 7.4), although north-west-trending structures tend to dominate, with a general thickening of units to the south-west. The thickening is very obvious on a section line perpendicular to the thickening orientation, refer to Figure 7.5. These observed thickness trends have been incorporated into the search orientations used for grade estimation.

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 7.4 KPS (Ki1.1.2) Vertical Thickness

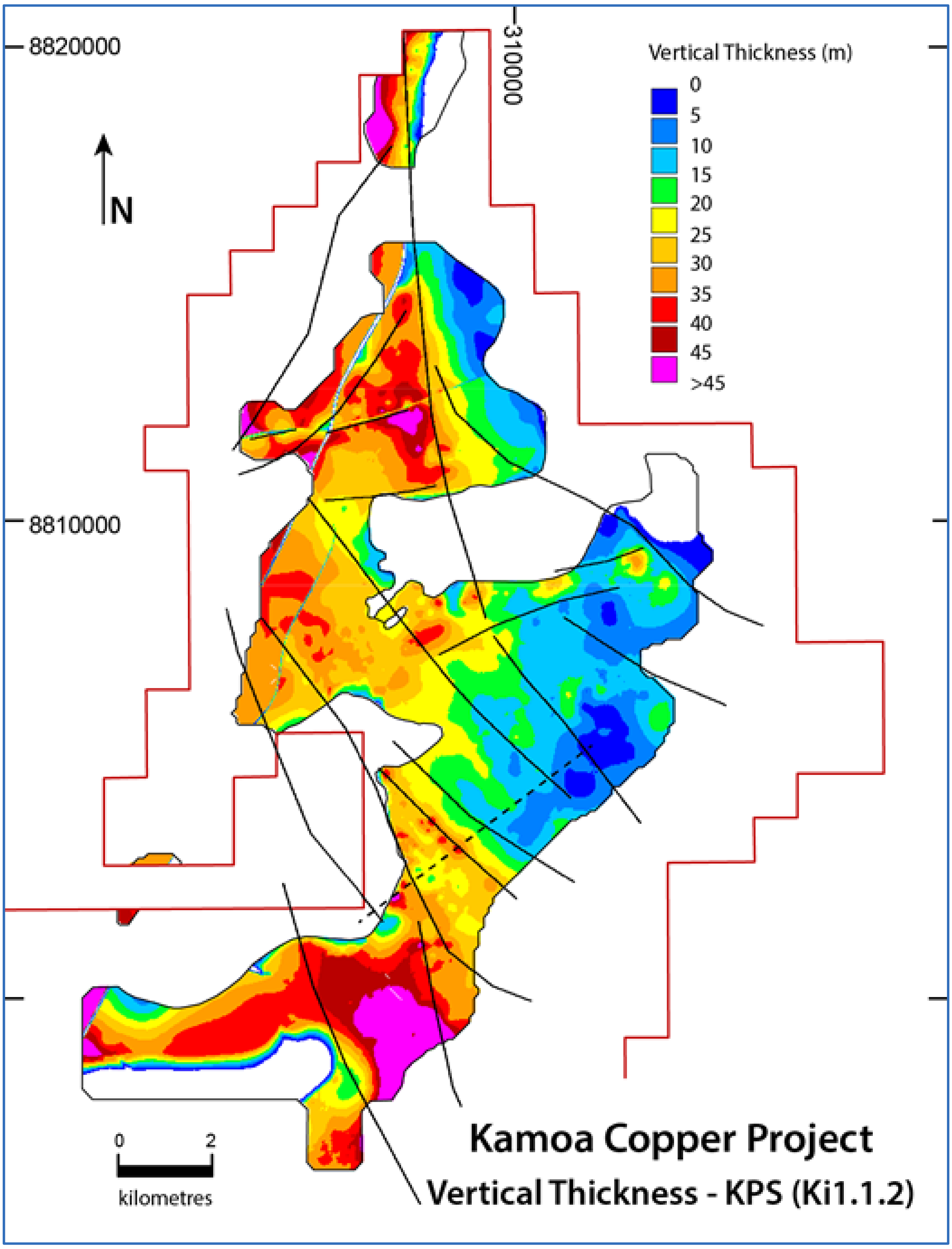


Figure provided by Ivanhoe, 2020; black lines are the interpreted growth fault positions; the trace of the cross-section shown in Figure 7.5 is shown in the dashed black line.

Figure 7.5 Section from Kansoko Sud (SW) to Kansoko Centrale (NW)

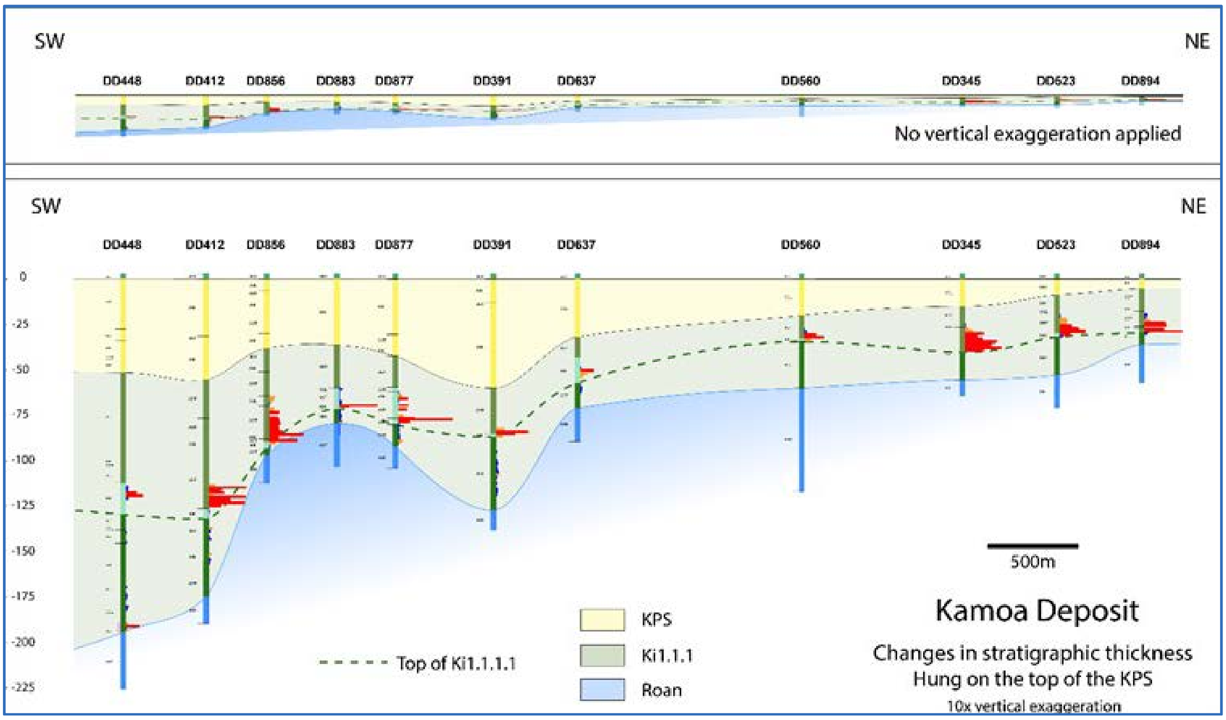


Figure provided by Ivanhoe, 2014, illustrating the thickening of units to the south-west; section line location is indicated by dashed line in Figure 7.4Figure 7.5. Copper grades are shown as histograms, with red being >1% TCu.

7.3.3 Structure

Geophysical data and topographic expression provide the primary support for regional continuity of structural features, whilst drillhole data and geotechnical logging provide local information to characterise more localised structures. Four major structures have been recognised, with the north-northeast-trending West Scarp Fault forming the primary brittle structure at Kamoa, with a west-side down-throw of approximately 200–400 m. These structures were used as boundaries to divide the mineralisation into structural zones, refer to Figure 7.6.

The presence of very open folds at Kamoa 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.

Microstructures are commonly observed in core, particularly in the finely laminated siltstone units. In rare cases, unusually steep bedding is identified to occur over intervals of 0.5–2 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.

Figure 7.6 Structural Model and Contours (masl) for the Roan-Ki1.1.1 Contact at the Kamoa Deposit

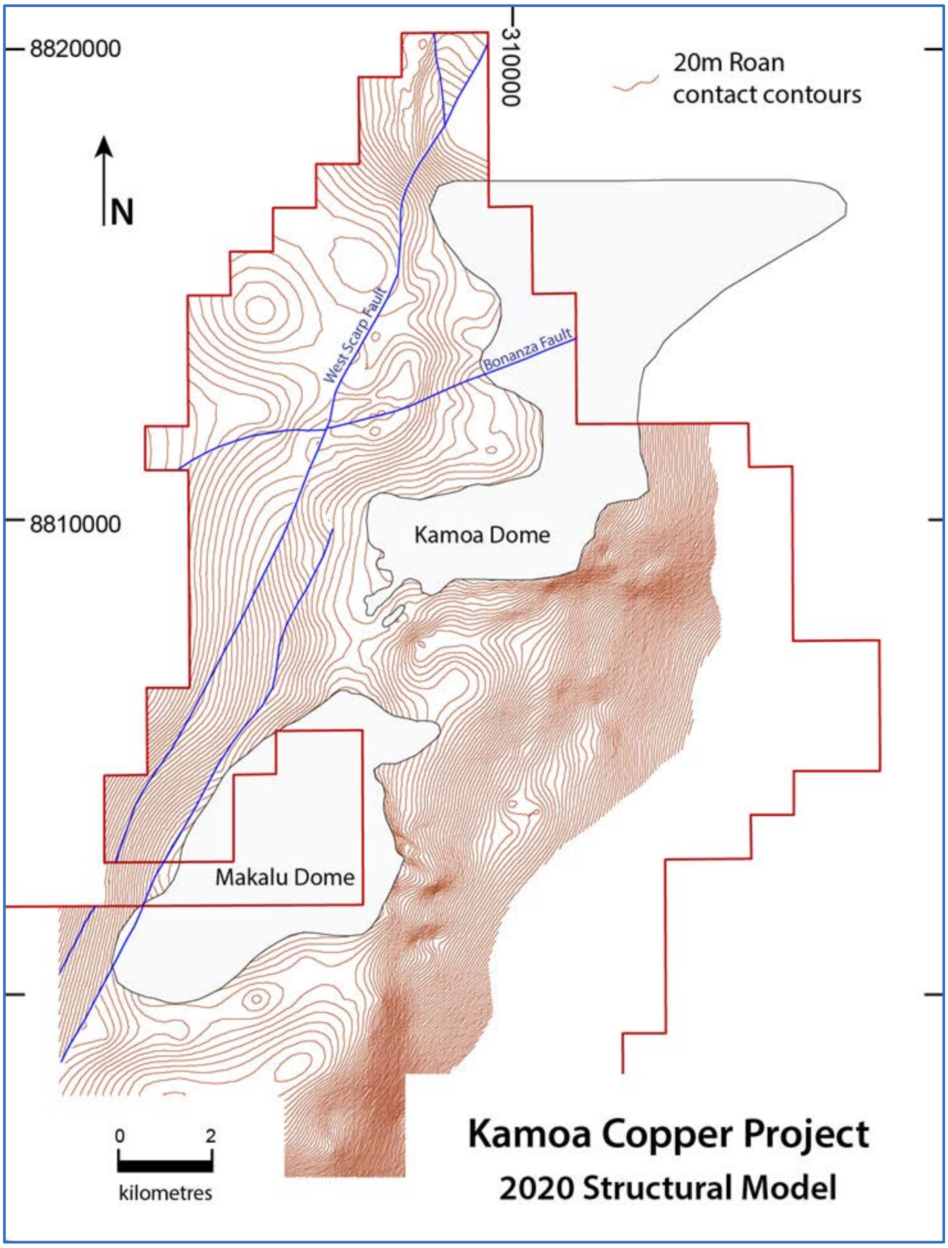


Figure provided by Ivanhoe, 2020.

7.3.4 Mineralisation

Mineralisation at Kamoa has been defined over an irregularly-shaped area of 24 km x 14 km. Mineralisation thicknesses at a 1.0% Cu cut-off grade ranges from 2.3–21.6 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 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.

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 moves consistently and predictably from one unit to another (Figure 7.7).

Figure 7.7 Stratigraphic Section Showing Continuity of Mineralisation Near Base of Ki 1.1.1.3 at the Kamoa Deposit (8807500N looking North)

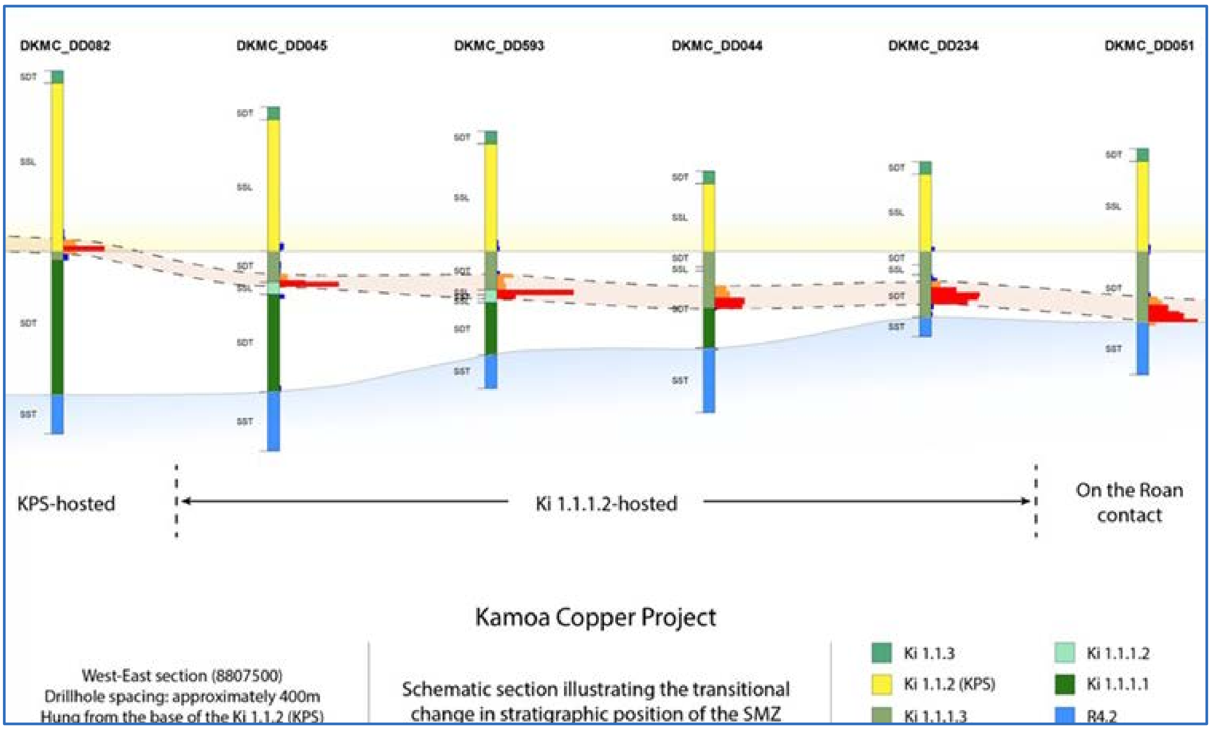


Figure provided by Ivanhoe, 2014. Copper grades in percent, shown as red histograms if > 1% TCu.

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 7.8).

Figure 7.8 Facies in Which Mineralisation Occurs

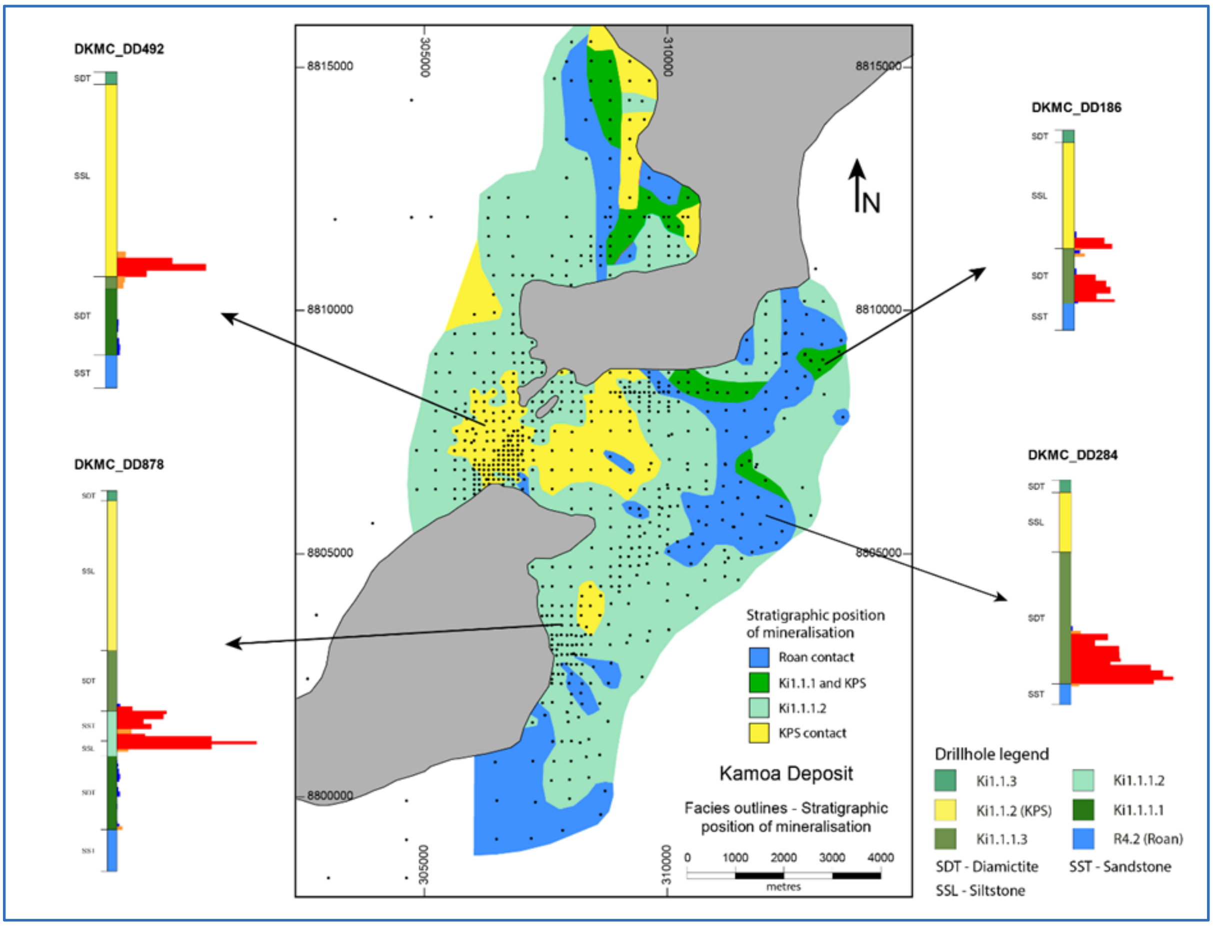


Figure provided by Ivanhoe, 2016. Copper grades in percent, shown as red histograms if > 1% TCu.

Foreland-hosted copper deposits such as the Kamoa deposit, typically show mosaic-patterns in terms of grade, thickness and stratigraphic position. Detailed drilling (spacing 100 m or less) will often show that these mosaic pieces can be on the order of a kilometre in extent with similar grades, thicknesses and stratigraphic positions. At their edges, there can be significant changes to grade or thickness over a few hundred metres. 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 sulfide concentration in the diamictite.

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 change from supergene to hypogene is generally transitional with a strongly developed vertical zonation evident in the hypogene.

At Kamoa, chalcopyrite is the primary sulfide mineral, and usually occurs as fine-grained disseminations in the diamictite matrix. However, very coarse-grained chalcopyrite can form as elongated grains up to 5 mm in length rimming clasts, or defining strain shadows to clasts. Bornite is typically fine-grained and disseminated in the matrix of the diamictite. When well developed, the fine-grained bornite is visually recognised 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 fine-grained chalcocite-dominant with secondary native copper and cuprite. The supergene zone may locally extend to depths of 250 m or more along fracture zones.

Since 2018, exploration has primarily focused on targets in the Kamoa North and Kamoa Far North regions. Within the Kamoa North region, a new style of mineralisation was discovered at the Bonanza Zone, where copper grades regularly exceed 20% TCu. These very high copper grades are believed to be the result of an east-west fault focussing copper-rich fluids to interface with both the typical mineralised horizon at Kamoa and the overlying, highly-sulfidic and reduced KPS (Ki1.1.2; refer to Figure 7.9). This has resulted in a stacked mineralised horizon, with the upper mineralised horizon hosted in the KPS (found in the vicinity of hole DD1450) and a lower horizon with typical diamictite-hosted mineralisation.

Drill sections on 50 m sections on strike in the central section, and 100 m apart elsewhere in the Bonanza Zone have shown that the very high grade mineralisation extends approximately 600 m along strike west of the West Scarp Fault, and 1,500 m along strike east of the West Scarp Fault. At a 1.0% Cu cut–off, the true thickness of the Bonanza Zone ranges from <1 m to 24.0 m (for Indicated Mineral Resources). The Bonanza Zone remains open to the west.

Drilling in the Far North Zone has defined 2,500 m of high-grade copper along an approximately north-south trending fault where fluids have been focussed into a very condensed sequence of basal diamictite and overlying KPS.

Figure 7.9 Section Showing the Copper Grades at the Kamoa North Bonanza Zone

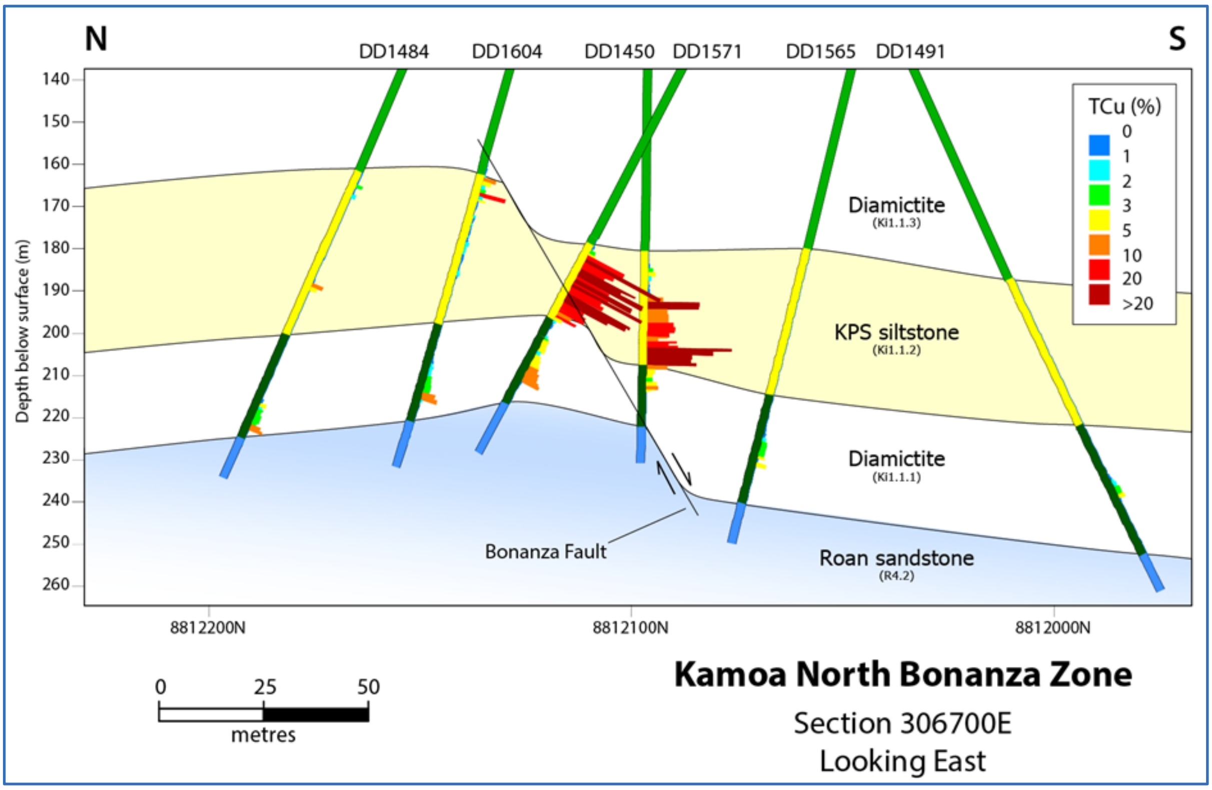


Figure provided by Ivanhoe, 2020.

7.4 Kakula Deposit

7.4.1 Lithologies

Sandstones of the Mwashya Subgroup of the Roan Group (R4.2) form the basal unit at Kakula. Kakula is located in an area where the basin has deepened, and the Ki1.1.1 package is significantly thickened. The distinction of clast-rich and clast-poor diamictites at Kakula is not as clear as at Kamoa. The diamictites of the Ki1.1.1 are generally clast poor and are typically silt-rich. Numerous siltstones are developed within the Ki1.1.1, especially in the lower half of the unit. Although these siltstones appear to be broadly continuous, 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. 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 shallowest portion of the Kakula deposit lies between the Kakula and Kakula northeast domes and dips less than 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.

7.4.2 Thickness of Modelled Units

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 7.10).

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 7.11). Locally the KPS has been entirely eroded where it crops out along the domes, and the thickness for the Ki1.1.1 could not be estimated. 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.

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

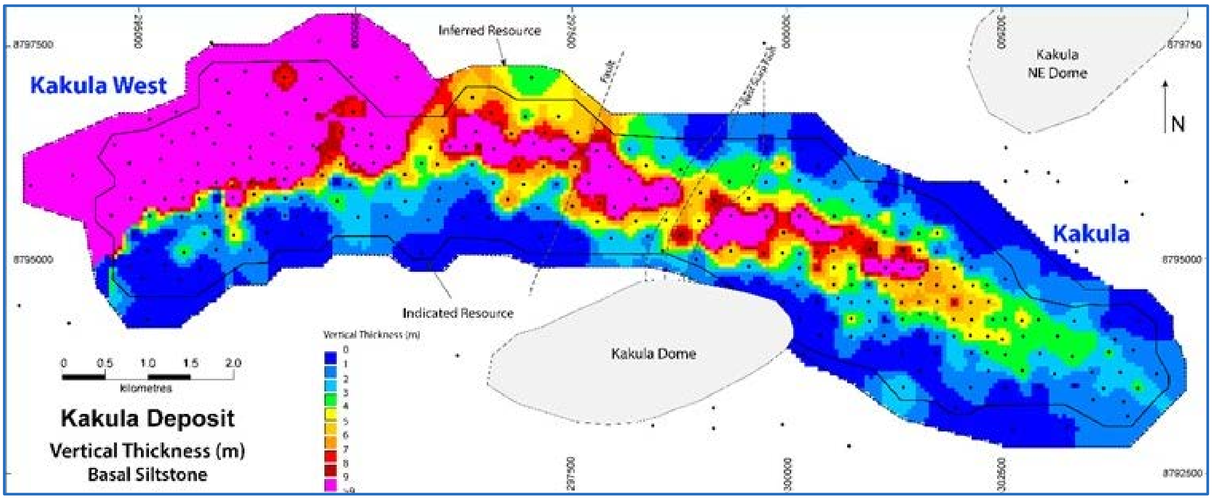


Figure provided by Ivanhoe, 2019. Vertical thickness estimated using an isotropic search.

Figure 7.11 Vertical Thickness of the Ki1.1.1 at the Kakula Deposit

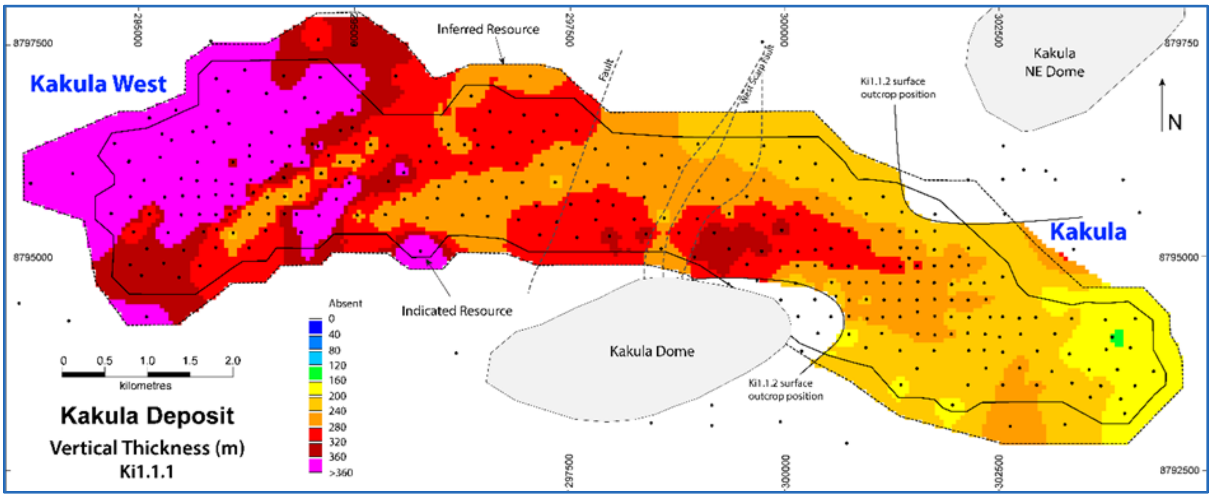


Figure provided by Ivanhoe, 2019. Vertical thickness estimated using an isotropic search.

7.4.3 Structure

The geometry of the Kakula deposit is strongly influenced by extensional faults. Because the faults were active during deposition, a number of sub-basins were formed across the axis of a broad doubly-plunging antiform, and lithological units appear to drape across the extensional faults rather than having discrete offsets. Extensional faults have been noted in the south east portion of the deposit, but do not appear well-developed.

At Kakula West, a series of sub-basins have been formed adjacent to extensional faults that strike north-east and east–north east. Draping of stratigraphic units over these extensional faults at the Ki1.1.1–R4.2 boundary can occur with elevation differences greater than 50 m. On the western edge of Kakula West, pronounced north east-trending extensional faults are evident, and elevation differences greater than 200 m (west block down) are observed in some areas.

Basin inversion associated with the Lufilian Orogeny 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 locally offset the mineralisation (Figure 7.12 and Figure 7.13). The most prominent of these faults trend north–north east and are probably related to the West Scarp Fault. Additional observed structures in drill core include steeply-dipping chaotic breccias and gouges, and cohesive “crackle” breccias.

Figure 7.12 Structure Model for the Kakula Resource Area Showing Contours (masl) for the Ki1.1.1–R4.2 (Roan) Contact

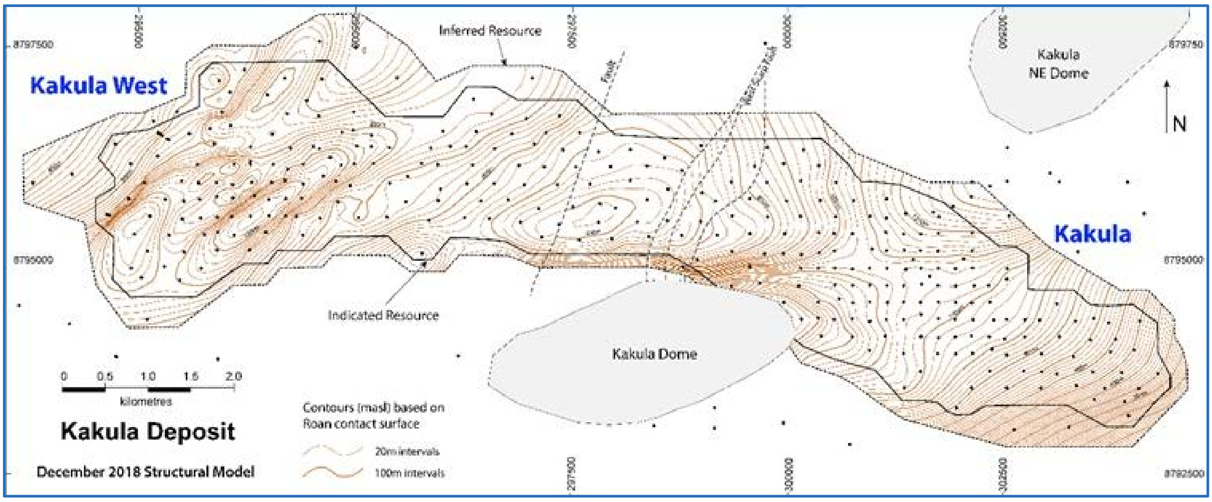


Figure provided by Ivanhoe, 2019.

Figure 7.13 Long Section of the North-West Kakula Area Illustrating Offset Across the Modelled Faults

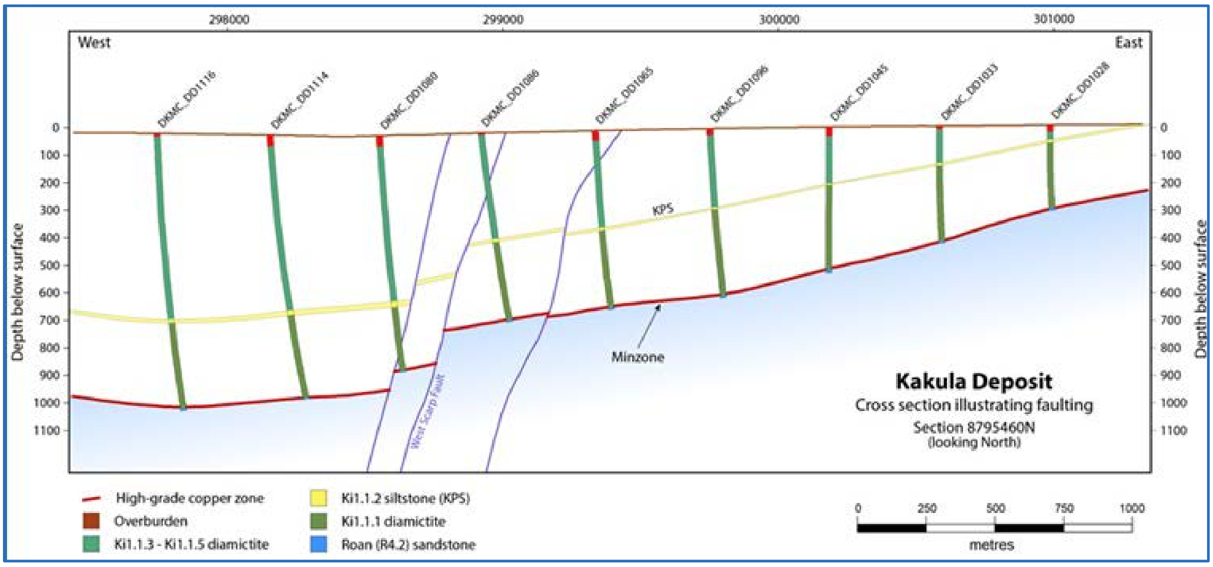


Figure provided by Ivanhoe, 2018.

7.4.4 Mineralisation

The Kakula deposit is currently delineated over an area of 14 km by 5km. The vertical thickness of the mineralisation at a 1.0% Cu cut–off grade ranges from 2.9 m to 42.5 m (in the indicated Mineral Resource area). The deposit has been tested locally from below surface to depths of more than 1,000 m, and remains open to the southeast and west.

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, contains very high grades (>6% Cu), and accounts for the majority of the deposit. The lateral continuity of this reductant allows for the unique lateral continuity of grades >6% TCu. 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 (Figure 7.14). 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 7.14 Northwest to Southeast Section Through Kakula Illustrating the Numerous Siltstone Units Developed Towards the Base of the Ki1.1.1

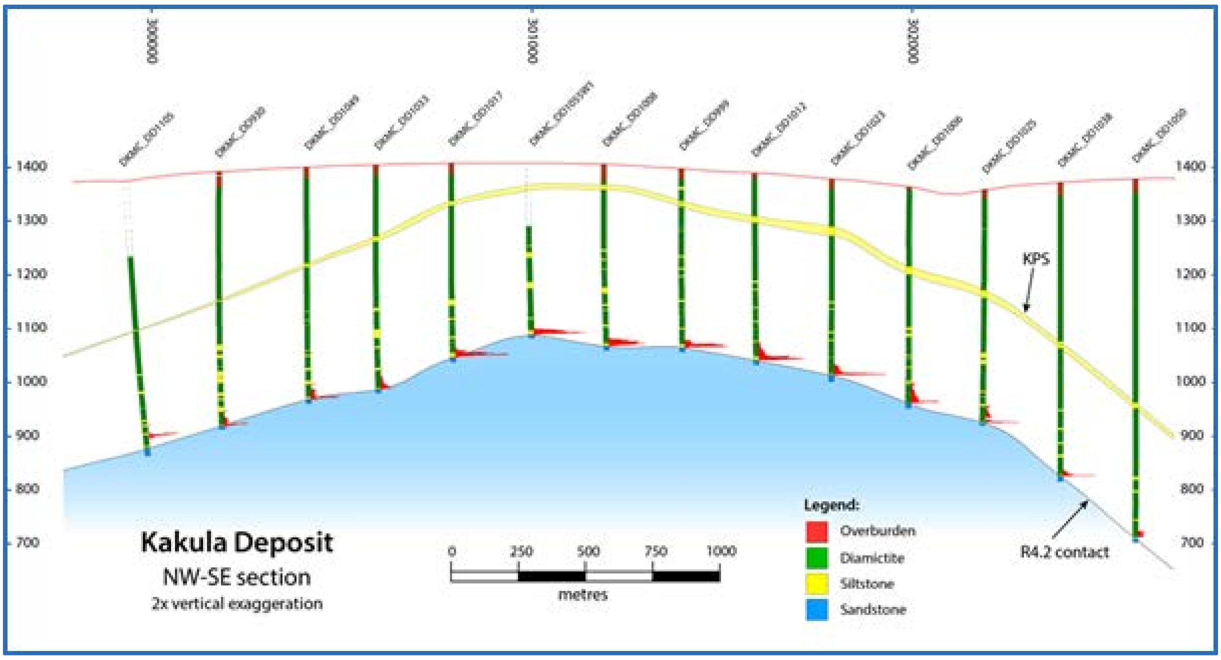


Figure provided by Ivanhoe, 2017. Red bars indicate assay intervals grading ≥ 0.5%.

Mineralisation at Kakula is dominantly hypogene chalcocite with gradual transition upward to bornite. Bornite and chalcopyrite zones are not as well developed as at Kamoa, and supergene chalcocite zones do not occur at Kakula. The chalcopyrite and bornite zones are very narrow, with a very gradual transition downward from bornite to chalcocite, followed by a zone that is typically within the basal siltstone, which is chalcocite-dominant (Figure 7.15). Whilst still dominantly fine-grained, numerous examples of coarse to massive chalcocite are evident in the highest-grade intersections. 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 7.15 Examples from Three Drillholes from Kakula of Vertical Mineral Zonation Evident Based on TCu:S Ratios

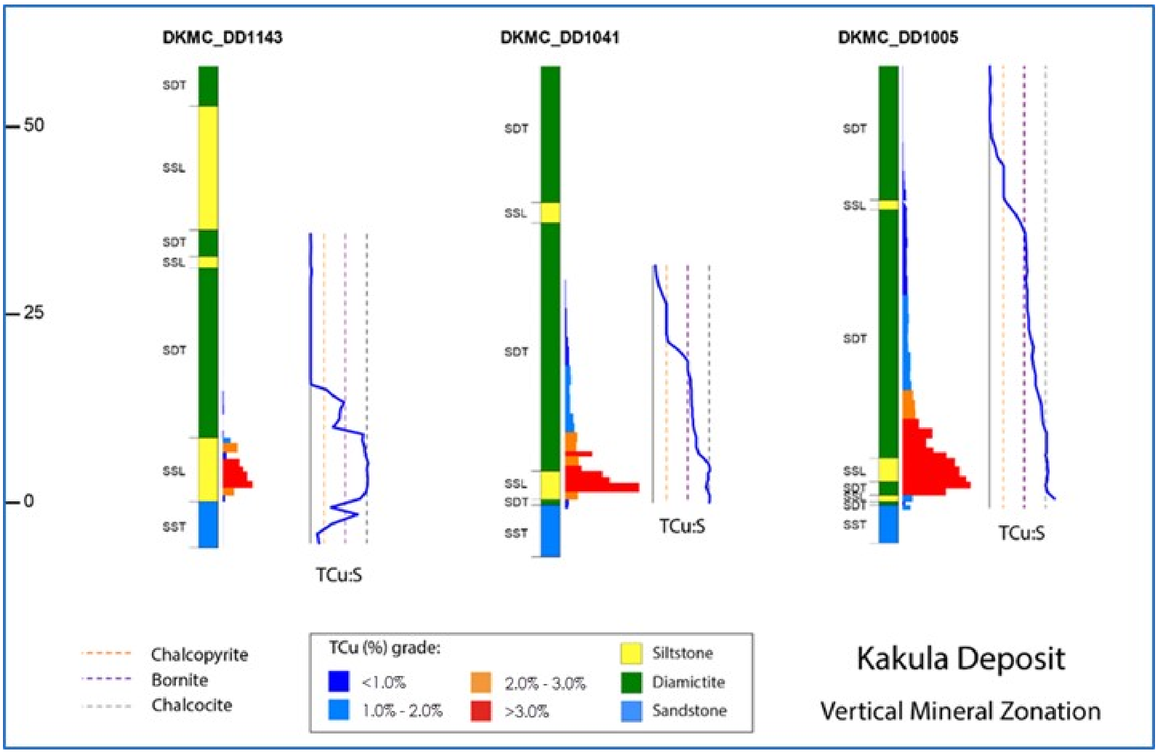


Figure provided by Ivanhoe, 2018.

In the south-eastern portions of Kakula, the highest-grade intersections trends 115° and aligns with the different stratigraphic and lithological units. To the northwest, 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 easterly orientations. The orientations of the controlling growth fault features has been incorporated into the search orientations used during grade estimation. The intensity of these controls and their incorporation into the grade estimation are discussed in Section 14.

7.5 Comments on Section 7

The Wood QP notes the following:

- Mineralisation within the Project has been defined over an irregularly-shaped area of 28 km x 23 km. The mineralisation is typically stratiform, and vertically zoned.
- The understanding of the deposit settings, stratigraphies, lithologies, structures, sulfide mineralogies, alteration and their controls on the mineralisation is well understood and sufficient to support estimation of Mineral Resources and Mineral Reserves at Kamoa, and Kakula.

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) sulfide-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, strata bound, and locally stratiform. Framboidal or colloform pyrite is common. Copper minerals typically replace pyrite and cluster around carbonaceous clots or fragments.
- Mineralisation timing: Sulfides and associated non-sulfide 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 sulfides 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 strata bound, 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 28 km x 23 km.
- Strong host-rock control and restriction of the mineralisation to a redox boundary zone between oxidised footwall haematitic sandstone and reduced, sulfidic 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 sulfide minerals from chalcocite to bornite to chalcopyrite.
- Hypogene minerals are chalcopyrite, bornite and chalcocite, with the predominant copper sulfide 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 sulfides 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 sulfidic clast rims in the basal diamictite and scattered tiny carbonate +/- quartz veinlets with occasional sulfides can occur at the Kamoa deposit.

Exploration programmes that use a strata bound, sediment-hosted model are considered applicable to the Project area.

9 EXPLORATION

9.1 Grids and Surveys

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.

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

Geophysical surveys completed over the Project are summarised in Table 9.1. The survey data are used in support of exploration vectoring in the Project area.

Table 9.1 Geophysical Surveys

Survey Type and Operator	Year	Comment
Airborne geophysics; Fugro Airborne Surveys (Pty.) Ltd	2004	Identified a number of magnetic lineaments that reflect underlying structures. One 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.
Downhole electromagnetic (EM); Gap Geophysics Australia and Quik_Log Geophysics	2011	Orientation survey in three holes at Kamoa. Included natural gamma, density, sonic, magnetic susceptibility, three component magnetics, resistivity, conductivity, induced polarisation and acoustic data (fractures).
EM orientation survey line	2011	Inconclusive results.
Ground magnetics	2011-2012	Used as a geology and structure mapping tool.
Ground gravity	2016	Eight lines at Kakula completed to help delineate the Ki1.1.1–R4.2 contact.
Downhole surveys; Quick Log Geophysics	2016–2017	12 drillholes. Logged full wave sonic, dual density, resistivity and gamma, collected acoustic televiewer (ATV) data.
2D seismic; HiSeis	2017–2018	Four regional scale lines completed to position the top of the Roan, interpret broad-scale basin architecture and locate both known and unknown growth and younger brittle structures.
Radiometrics (Excalibur); ground gravity and ground magnetics (Ivanhoe)	2019	Airborne radiometric surveys were completed over the planned Kakula tailings storage facility footprint. Ground gravity, ground magnetics and airborne radiometrics were conducted in the Kamoa North area to better understand the controls of the very-high-grade mineralisation.

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. Results indicated possible K_2O enrichment commensurate with potassic (feldspar–sericite) alteration.

A MSc thesis was completed in 2013 on the Kamoa stratigraphy, diagenetic and hydrothermal alteration, and mineralisation. An accompanying paper has been published in *Economic Geology* (Schmandt, et al, 2013).

Two additional studies have been summarised 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 localising high grades (Twite et al., 2019), and the origin of the thick diamictite packages as subaqueous debris flows (rather than primary glacial 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 Conglomérat 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.

The eastern boundary of the Mineral Resource estimate at Kamoa is defined solely by the current limit of drilling, at depths ranging from 600–1,560 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.

At Kakula, the western and south eastern boundaries of the high-grade trend within the Mineral Resource estimate area are defined solely by the current limit of drilling and the deposit remains open in these directions.

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

The Wood QP notes:

- 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 within the Project area.

10 DRILLING

10.1 Introduction

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 18 September 2020, there were 2,159 coreholes completed (Table 10.1). Collar locations are provided in Figure 10.1.

The drillhole database used for the Kamoa resource estimation was closed on 20 January 2020. The 2020 Kamoa Mineral Resource estimate used 998 drillhole intercepts. Included in the 998 drillholes are 17 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 drillhole database used for the Kakula resource estimation was closed on 1 November 2018. The November 2018 Kakula Mineral Resource estimate used 354 drillhole intercepts.

The 807 holes not included in either the Kamoa or Kakula estimates were excluded because they were either abandoned, unmineralised holes in the dome areas, unsampled underground cover, metallurgical, civil geotechnical or hydrological drillholes, or were drilled after the closure of the various databases for estimation purposes.

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 18 September 2020)

Drill Purpose	Count (Active)	Metres (m)
Kamoa estimate (Jan 2020)	998	288,140.7
Kamoa (post-estimation)	79	17,498.8
Kakula estimate (Nov 2018)	354	196,549.8
Kakula (post-estimation)	85	35,014.8
Exploration	7	2,677.9
Domes	107	7,856.3
Metallurgy	132	13,996.2
Geotechnical	57	7,757.7
Civil Geotechnical	99	2,787.6
Condemnation	51	1,177.8
Cover Drilling	62	8,947.3
Hydrogeology	11	781.9
Abandoned	117	26,635.6
Total	2,159	609,822.2

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

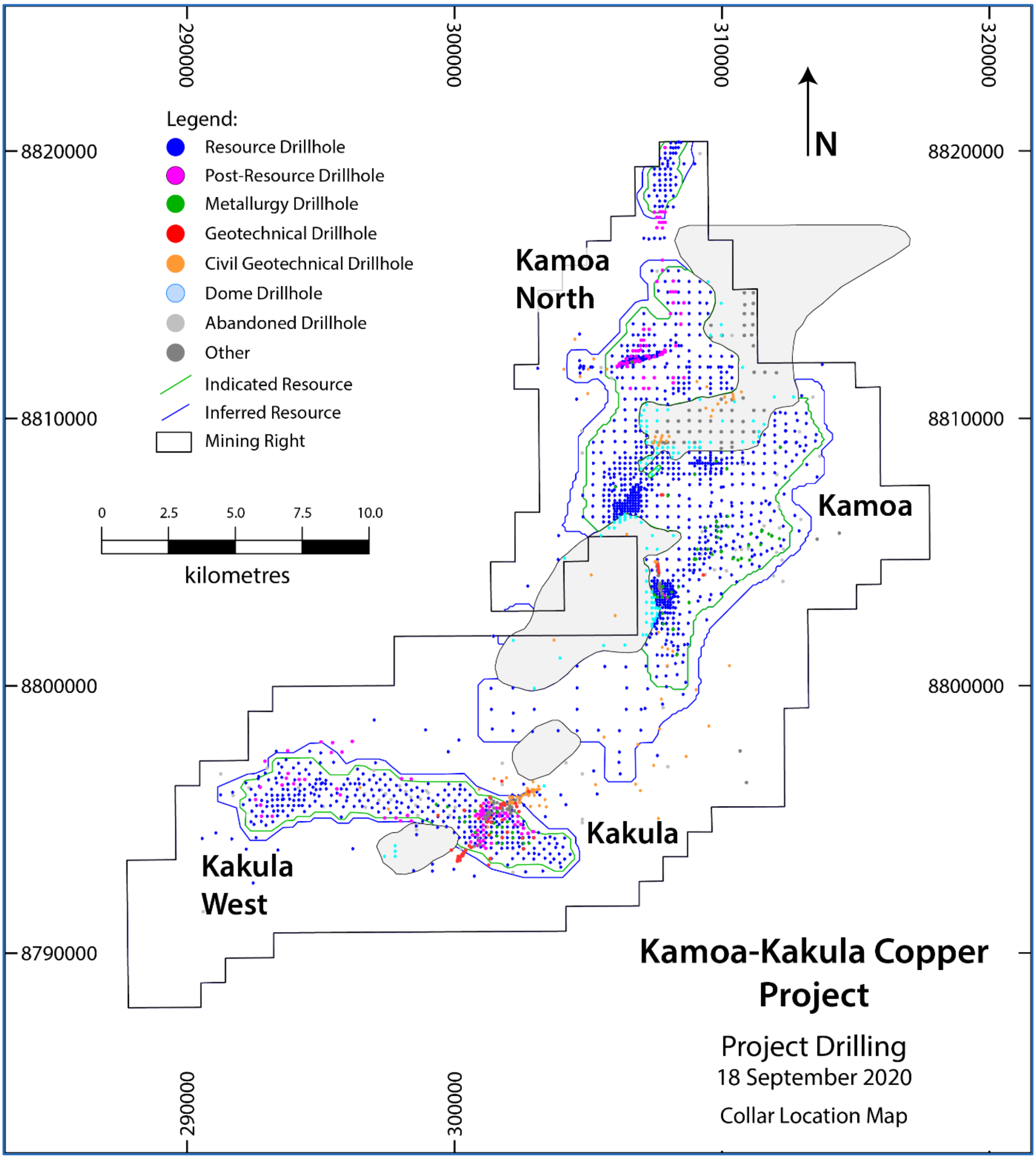


Figure provided by Ivanhoe, 2020. 'Other' includes exploration drillholes, condemnation drillholes, cover drillholes and permeability drillholes.

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.

Prior to 2012, 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 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.

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

10.4 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.

10.4.1 Kamoa

As of 10 January 2020, only three drillholes (DKMC_DD1580, DKMC_DD1600 and DKMC_DD1621) lacked an independently surveyed collar position. In these cases, the planned coordinate positions are used.

10.4.2 Kakula

All collars for holes used in the Kakula Mineral Resource estimate were independently surveyed.

10.5 Downhole Surveys

10.5.1 Kamoa

Corehole orientations ranged from azimuths of 0° to 360°, with downhole inclinations that ranged from -5.0º to vertical. Most holes were vertical or subvertical, with only the geotechnical drillholes (-45º) and cover drillholes (<-10º) 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 2020 drillholes, using a Single Shot digital downhole instrument. Once the hole was completed, a Reflex Multi Shot survey instrument was 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.

10.5.2 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.

10.6 Geotechnical Drilling

Ivanhoe collects geotechnical and structural information from resource drillholes. Rock mass characterisation has been carried out on 1,203 drillholes and used in the geotechnical investigation. Samples were collected for laboratory testing of intact rock strength properties from dedicated geotechnical drillholes or separate wedges drilled from resource drillholes. Details are provided in Sections 16.1 and 16.2.

10.7 Metallurgical Drilling

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

10.8 Drilling Since the Mineral Resource Database Close-off Date

10.8.1 Kamoa

The database contains 79 drillholes (17,498.8 m) that post-date the Kamoa resource estimate database close-off date of 20 January 2020 (Figure 10.2). These holes were drilled for resource purposes, either as infill drillholes, or resource expansion drillholes, or for geotechnical and metallurgical purposes in the Kamoa North Bonanza Zone area.

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 Wood QP considers that the new drilling should have no material effect on the overall tonnages and average grade of the current Mineral Resource estimate.

10.8.2 Kakula

Between 1 November 2018 and 18 September 2020, Ivanhoe completed an additional 85 core drillholes (35,014.8 m) at Kakula. The collar locations of the coreholes are shown in Figure 10.2. The core drillholes were drilled for exploration and infill purposes.

New holes within the existing Indicated Mineral Resource estimate area are geotechnical or infill drillholes in close proximity to current underground development that generally show similar grades as the resource model, and the Wood QP considers that this new drilling should have no material effect on the overall tonnages and average grade of the Indicated Mineral Resource. The 13 new drillholes outside the Indicated Mineral Resource model, however, may slightly increase the tonnage of the Inferred Mineral Resource and have upside potential for Mineral Resource estimation when incorporated into an updated model.

Figure 10.2 Plan View Showing Kamoa-Kakula Drillholes Completed Since Construction of the Respective Mineral Resource Models (at 1 March 2019)

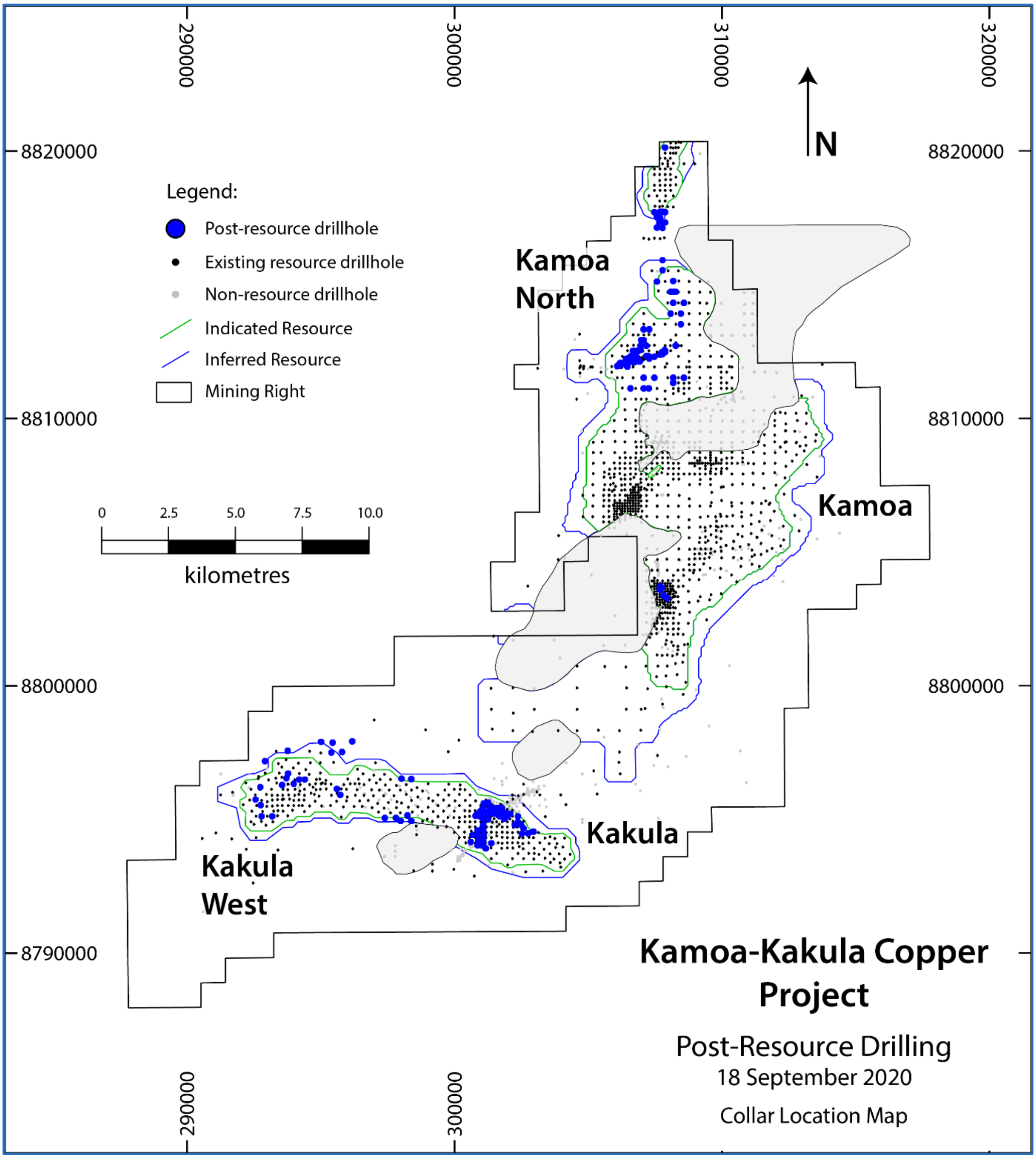


Figure provided by Ivanhoe, 2020.

10.9 Comments on Section 10

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. The Wood QP notes:

- Examples of summary results and interpretations of drilling are illustrated in Figure 7.5, Figure 7.7 to Figure 7.9, Figure 7.13 to Figure 7.15, and Figure 14.10.
- 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 intercept selected as the “selective mineralised zone” can include both lower and higher-grade mineralisation; however, the transition in grade from non-mineralised to >1% Cu is usually distinct. 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

11.2.1 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 (30-40 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.

11.2.2 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.

11.2.3 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 KPS (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 hanging wall, and the prepared pulp was analysed by Niton. The results are used to characterise the geochemistry of the hanging wall material.
- After August 2014, whole core was logged by the geologist on major lithological intervals, until mineralised material or at a “zone of interest” (ZI) such as a lithology that is conventionally sampled (e.g. the KPS) was encountered. Note that the KPS is not routinely sampled at Kakula, as it occurs >100 m above the mineralised zone. The ZI was logged on sampling intervals, typically 1 m intervals (dependent on geological controls). Within any zone of interest, the geologist highlighted material that was either mineralised or material that was expected to be mineralised. This “zone of assay” (ZA) was extended to 3 m above and below the first sign of visible mineralisation.
- 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 was halved for sampling purposes using an automated core cutter with diamond saw. 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 were split using a palette knife.

11.3 Metallurgical Sampling

11.3.1 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.

11.3.2 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 programmes.

11.4 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. There is a total of 28,520 SG measurements were performed on samples taken from drill core. Of these measurements, there are 28,503 samples with SG values between 1.5 and 4.0.

For Kakula, all samples selected for copper analysis (from DKMC_DD1002 onwards) are also measured for SG using the entire sample interval.

There is a total of 17,697 SG measurements that 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 17,685 samples with SG values between 1.5 and 4.0.

11.5 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 2020 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

11.6 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 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 (CRMs) and blanks are included with the sample submissions.

11.7 Sample Analysis

In this report, two forms of copper assay methods are reported: total copper (TCu) and sulfuric acid soluble copper (ASCu).

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.

Bureau Veritas acquired Ultra Trace in 2007. As the assay certificates for Kamoa were certified by Ultra Trace, Wood 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% sulfuric 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. ASCu analyses were stopped during the Kakula drill programme. Drilling was still ongoing at Kakula when drilling recommenced at Kamoa and so 19 of the earlier drillholes from this period (DKMC_DD1172 to DKMC_DD1339W1) also lack ASCu data. The vast majority of holes for Kamoa from DKMC_DD1372 onwards have ASCu data.

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), although Ag analyses were discontinued in 2019.

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% sulfuric 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.

11.8 Quality Assurance and Quality Control

Quality assurance and quality control (QA/QC) samples are placed using between 5% and 7% insertion rate for CRMs, blanks and duplicates within the ZA, and between 3% and 5% for the ZI. There are always at least two original samples before any new QA/QC insertion.

11.8.1 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.

11.8.1.1 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.

11.8.1.2 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, was inserted after sample preparation as it was 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.

11.8.2 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.

11.8.3 Certified Reference Materials

Kamoa uses CRMs sourced from independent companies, Geostats Pty Ltd (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 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.

For the Kamoa North drill programme, nine matrix-matched and three commercial CRMs were used 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 were certified by Mr. Dale Sketchley, P. Geo. of Acuity Geosciences (Acuity). Commercial CRMs were purchased from OREAS, and AMIS.

Kakula uses six matrix-matched and commercial CRMs to monitor the accuracy of assay performance. Matrix-matched CRMs were created and certified using the same procedure described for Kamoa. Commercial CRMs were purchased from OREAS, and 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.

11.9 Databases

In early 2013, Ivanhoe implemented an acQuire data management database for storage of all relevant electronic data. Ivanhoe and Acuity have completed validations to ensure the data integrity was 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.

Where they exist from older drill programmes, 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.

11.10 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.

11.11 Sample Storage

Half and quarter core reference samples are stored in metal trays in a purpose-designated core storage shed. The core storage comprises four lockable buildings with 24-hour security personnel in place. A fifth storage facility has been constructed for storage of the Kakula drillholes.

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 Wood QP, 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 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.

12 DATA VERIFICATION

12.1 Wood Verifications (2009-2020)

Between 2009 and 2020, Wood 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). An audit in support of the current resource update was completed in February 2020 (Reid, 2020).

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.

12.2 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 sulfides 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.

12.3 QA/QC Review

As part of the data verification above, Wood reviewed the QA/QC data or QA/QC reports to ensure the assay data were of sufficient quality to support Mineral Resource estimation.

Wood personnel 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 Wood. QA/QC verification is summarised in Table 12.1.

Table 12.1 QA/QC Verification

Review Type	Duration	Comment
Screen tests	2009–2013	Conducted by both the sample preparation facility on-site and by Ultra Trace. 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 re-pulverisation shows results of over 90% passing 75 µm were achieved.
CRMs	2009–2013	Sample submissions included packets of CRMs purchased from commercial vendors OREAS, AMIS and Geostats. The primary CRMs are from OREAS and AMIS. 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.
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.
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%. The assay precision is acceptable for Mineral Resource estimation.
Blanks	2009–2013	Results for 1,882 blank samples were reviewed. Ivanhoe concluded that the blank material has low concentrations of the target elements Cu and Co (Carver, 2009a). Though the results indicate to Wood 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 Wood considers that there is no significant risk to the Mineral Resource estimate.
Kamoa QA/QC review	2014	Ivanhoe submitted 13 CRMs, blanks, and coarse reject duplicates as part of their QA/QC programme as summarised in Acuity (2014). Wood concluded that coarse reject duplicate results indicate adequate precision, blank samples do not indicate any sample contamination, and CRM results do not indicate any biases greater than 5%.

Review Type	Duration	Comment
Kakula QA/QC Review	2016–2018	Up to 20 May 2017, Ivanhoe submitted nine commercial and six matrix-matched CRMs, blanks, and coarse reject duplicates. Subsequent to 20 May 2017, four commercial and six matrix-matched CRMs have been submitted. QA/QC was reviewed by Acuity (2017, 2018a, 2018b, 2018c, 2018d). Wood concluded that 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. The effect of this one failure on the overall quality of data is not material. CRM results indicate biases much less than 5%.
Kamoa QA/QC Review	2020	Acuity assessed data from 121 drillholes at Kamoa up to 31 December 2019 (Acuity, 2020). The failure rate for CRMs and blanks inserted at the project site was very low and the documented failures were adequately addressed by re-assaying. Duplicate data compare well indicating a high degree of repeatability. 170 samples from the Kamoa North Bonanza Zone were above 15% TCu. At these elevated grades, an independent check assay is required. Sample selection for the check assays is in progress.

A number of check assay programmes were conducted. In each case, samples were selected to be representative of five copper grade populations based on natural breaks: extreme >15%; main >6.5%; lower >2.5%; halo >1.0%; and background >0.25%. All samples were submitted to ALS Vancouver, where they were subject to the same digestion method as Bureau Veritas. ALS Vancouver used a sodium peroxide fusion.

The initial programme consisted of a set of 196 representative routine samples from 50 drillholes completed between June 2009 and August 2016. 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 values.

A total of 277 samples was selected from 73 Kakula drillholes completed between August 2016 and May 2017 (Acuity, 2018a). A total of 20 matrix-matched CRMs, 15 blanks, and 10 pulp duplicates were inserted.

A total of 356 samples were selected from 130 drillholes completed at Kakula between May 2017 and January 2018 (Acuity, 2018c). 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 check sample assay programmes conducted by ALS Vancouver laboratory validated the original Bureau Veritas copper assays within a normally-expected range of laboratory variations.

12.4 On-Site Visits

The Wood QPs visited the Project as outlined in Section 2.4. Table 12.2 summarises the data validation and checks performed during the site visits.

Table 12.2 On-Site Data Verification

Review Type	Dates	Comment
Field drill collar check	2009, 2010, and 2011, 2012, 2016, 2017, 2020	Used a hand-held GPS unit to check selected field drill collar locations. No errors were noted in the collar surveys, and all results were within the error margin of a hand-held GPS.
Drill core inspection	2009, 2010, 2011, 2012, 2016, 2018, 2020	Logging details were noted, in general, to match the features that Wood staff, including the Wood QP, observed in the inspected cores. The identification of lithological units, alteration and sulfide mineralogy was considered by the Wood QPs to be appropriate to provide support for resource modelling.
Sample preparation facilities	2009, 2010	The sample preparation facilities operated by African Mining Consultants in Kolwezi and supervised by Richard Carver were inspected. No material issues were noted.
Sample preparation facilities	2011, 2012, 2016, 2017	Inspected the Kamoa-site sample preparation facility. No material issues were noted.
Underground inspection	2020	Inspected underground connection drives at Kakula that have exposed the basal siltstone and chalcocite-dominant mineralisation.

12.5 Copper Grade Witness Sampling

Wood conducted copper grade witness sampling in 2009, 2010, 2011, 2012, 2016, and 2017. The results of these check programmes are provided in Table 12.3.

Table 12.3 Copper Grade Witness Sampling

Date	Number of Samples	Laboratory	Comment
2009	21 half-core sample intervals from Kamoa were re-sawn, and quarter-core samples taken. Submission included CRMs and blanks	SGS Lakefield, Canada. ISO 17025-certified; independent of Ivanhoe	The correlation between the laboratories was good. The ratio of the mean Ultra Trace to SGS assays for Cu was 1.01.
2010	22 half-core sample intervals from Kamoa were re-sawn, and quarter-core samples taken. Submission included CRMs and blanks	ALS Vancouver	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.
2011 (Feb)	11 half-core sample intervals from Kamoa were re-sawn, and quarter-core samples taken. Submission included CRMs and blanks	Ultra Trace	Assayed for total copper and minor elements. These results from Ultra Trace were compared to the original Ultra Trace results, and found to be acceptable.
2011 (Nov)	Eight half-core sample intervals from Kamoa were re-sawn, and quarter-core samples taken. Submission included CRMs and blanks	Ultra Trace	Comparable to the original Ultra Trace results; Wood's Cu results were 4% lower than the original assays, while the ASCu results were 2% higher.
2012	11 half-core sample intervals from Kamoa were re-sawn, and quarter-core samples taken. Submission included CRMs and blanks	Ultra Trace	Ultra Trace's check sample results averaged 10% lower than the original Ultra Trace assays.
2016	Four half-core sample intervals from Kakula were re-sawn, and quarter-core samples taken. Submission included CRMs and blanks	Bureau Veritas	Confirmed the presence of copper mineralisation.
2017	20 half-core sample intervals from Kakula were re-sawn, and quarter-core samples taken. Submission included CRMs and blanks	Bureau Veritas	The check sampling showed good correlation with the original assays and a ratio of check/original of 1.02 for total copper.

12.6 Comments of Section 12

The Wood QP considers that the data verification programmes undertaken on the core data collected from the Kamoa and Kakula deposits 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 at Kakula.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Testwork Overview

The Kamoa (Kamoa/Kansoko) 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 discovered the Kakula deposit, which has significantly higher copper head grades, compared to the Kamoa deposit. Metallurgical testwork on the Kakula deposit was initiated in 2016 with most of the subsequent focus since 2016 on the Kakula deposit.

This report provides a summary of the historical testwork completed on the Kamoa deposit for the Kamoa prefeasibility study and development of the IFS4a flow sheet. The metallurgical testwork conducted on Kakula material for the Kakula prefeasibility and feasibility studies is described in detail.

13.1.1 Metallurgical Testwork on the Kamoa Resource

Between 2010–2015 a series of metallurgical testwork programmes, defined as Phases 1–5, 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 ore body 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 orebody, 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.

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 and 2015 at Xstrata Process Support (XPS) Laboratories. The Phase 6 samples best represent ores to be processed according to the early years of the Kansoko PFS mine schedule. It is noted that many of the Phase 2 and Phase 3 samples are relevant to the current Kansoko 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 3.96–8.19%.

Mineralogical work conducted by XPS in September 2016 indicated that the main Cu sulfide 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 Kakula PFS testwork campaign, which focussed on flow sheet optimisation as part of the Kakula 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 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 sulfide 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 9.8–13.5 kWh/t, characterising the material as soft with regards to crushing energy requirements. The abrasion index values ranged from 0.01–0.06 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 siltstone 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–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 287 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. A grindmill test is a batch milling test used to determine breakage and selection function parameters to aid in mill design. 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 5–8% 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^{2}h. 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 Flotation Variability Testwork

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

The samples tested varied from 2.6–9.2% Cu, with sulfur 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 sulfide Cu at 86%.

The Cu sulfide 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 sulfides. The average Cu sulfide grain sizes varied significantly from 8–20 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-sulfide 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-sulfide Cu are not consistent across the Kakula ore body. 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 were 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 Additional Metallurgical Testwork on Kakula Resource

Following the completion of the prefeasibility study, further testwork was initiated in March 2019 as part of the feasibility study, and consisted of:

- A mini-pilot plant campaign including Jameson Cell testwork, conducted by XPS.
- Desliming cyclone testwork, conducted by Multotec, South Africa.
- Flocculant screening testwork, conducted by ChemQuest, South Africa.
- Various slimes and full tailings settling testwork, conducted by Outotec, Paterson & Cooke, and Andritz.
- Concentrate regrind hydro cyclone and signature plot testwork, conducted by Grinding Solutions.
- Flotation tests utilising underground mine water, conducted by XPS.

Mineralogical assessment on the MPP sample indicated that the sample's mineralogy was similar to the PFS development composite sample and contained 12% Cu sulfide which consisted mainly of chalcocite (89%) and bornite (8.8%).

Duplicate open circuit cleaner tests were performed to baseline the MPP composite against the PFS flow sheet without any modification to reagent dosages, which reported a rougher grade and recovery in line with the PFS results, however, the scavenger cleaner circuit reported higher Cu losses. The final concentrate Cu recovery was noted as 79.6% at 64.3% Cu and 8.9% SiO_2. Another open circuit cleaner test was conducted during which the reagent dosage was increased to cater for the higher sample head grade. The adjustment in reagent dosing resulted in a final recovery of 85.6% Cu at a final product grade of 57.3% and 14.9% SiO_2.

A single locked cycle test was conducted to determine the effect of recirculating the scavenger recleaner tailings back to the scavenger cleaner. The circuit reached and maintained stability quickly once the recirculating loads were established. A total recovery of 82.2% Cu, at a final product grade of 63.6% Cu and 9.9% SiO_2 was recorded. Cu lost to the rougher/scavenger tailings was noted as 8%, and in line with the open circuit tests. The Cu losses to the scavenger cleaner tailings was slightly lower compared to the open circuit test (9.8% compared to 11.5%). Overall, the locked cycle test increased the Cu recovery by 2.6%, compared to the open circuit tests, at an increase of 1% SiO_2 grade in the final product.

Rougher concentrate product produced during the third MPP run was used to demonstrate the scale up of the low entrainment cleaning during bench scale testing, to the performance using a pilot Jameson Cell unit. The high-grade cleaner Jameson cell test compared well against the benchmark set in the open circuit tests. This single test indicated that the single stage Jameson cell performance will be able to match the results produced in the three stage bench scale dilute cleaning tests. A further Jameson cell test was conducted on scavenger cleaner concentrate product to investigate the need for concentrate regrind and scaling of the Jameson cell. The scavenger re-cleaner Jameson cell upgraded cleaner scavenger concentrate – not subjected to regrinding – from 18.1% Cu to 31.9% Cu, recovering just under 90% of the Cu. The first increment of concentrate achieved a 48.7% Cu grade. It was noted that the Jameson cell run without regrind matched the open circuit test which excluded the regrind step. Further, the exclusion of the regrind step resulted in a much lower product grade and recovery. It is not recommended to process the Kakula material without the regrind step.

During the flocculant screening and tailing thickening campaigns, it was noted that a tailings thickening circuit, designed at a flux of 0.42 t/h/m^2 could produce an underflow product of 57% solids (w/w) when dosing 30 g/t SNF 45 VHM and 60 g/t SNF 910 SH, with an overflow clarity of <100 mg/l. Dosing of a coagulant is required to maintain a clear overflow product.

The signature plot testwork reported an energy requirement of 20.14 kWh/t for the regrind step to achieve a combined product of P_{80} 10 µm.

The open circuit flotation testing utilising tap water and mine water yielded similar Cu recovery and grades.

13.1.6 Metallurgical 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 sulfide 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.

13.1.7 Kamoa Sample Performance on Kakula Flow Sheet

In 2018 XPS tested the performance of the Kamoa Phase 6 signature plot composite sample 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 have a similar metallurgical response and that the selected concentrator flow sheet is common concentrator to both.

13.2 Metallurgical Testwork on the Kamoa Resource

Between 2010–2015 a series of metallurgical testwork programmes 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 1–5, 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 (Year-1 to Year-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 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 Kamoa Metallurgical Sample Locations

The drillhole locations that provided the historical Kamoa Phase 1–5 metallurgical samples. 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 1–5 samples only are shown in Figure 13.1.

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 hanging wall and footwall samples were sourced and subjected to testing.

Figure 13.1 Kamoa Metallurgical Sample Locations

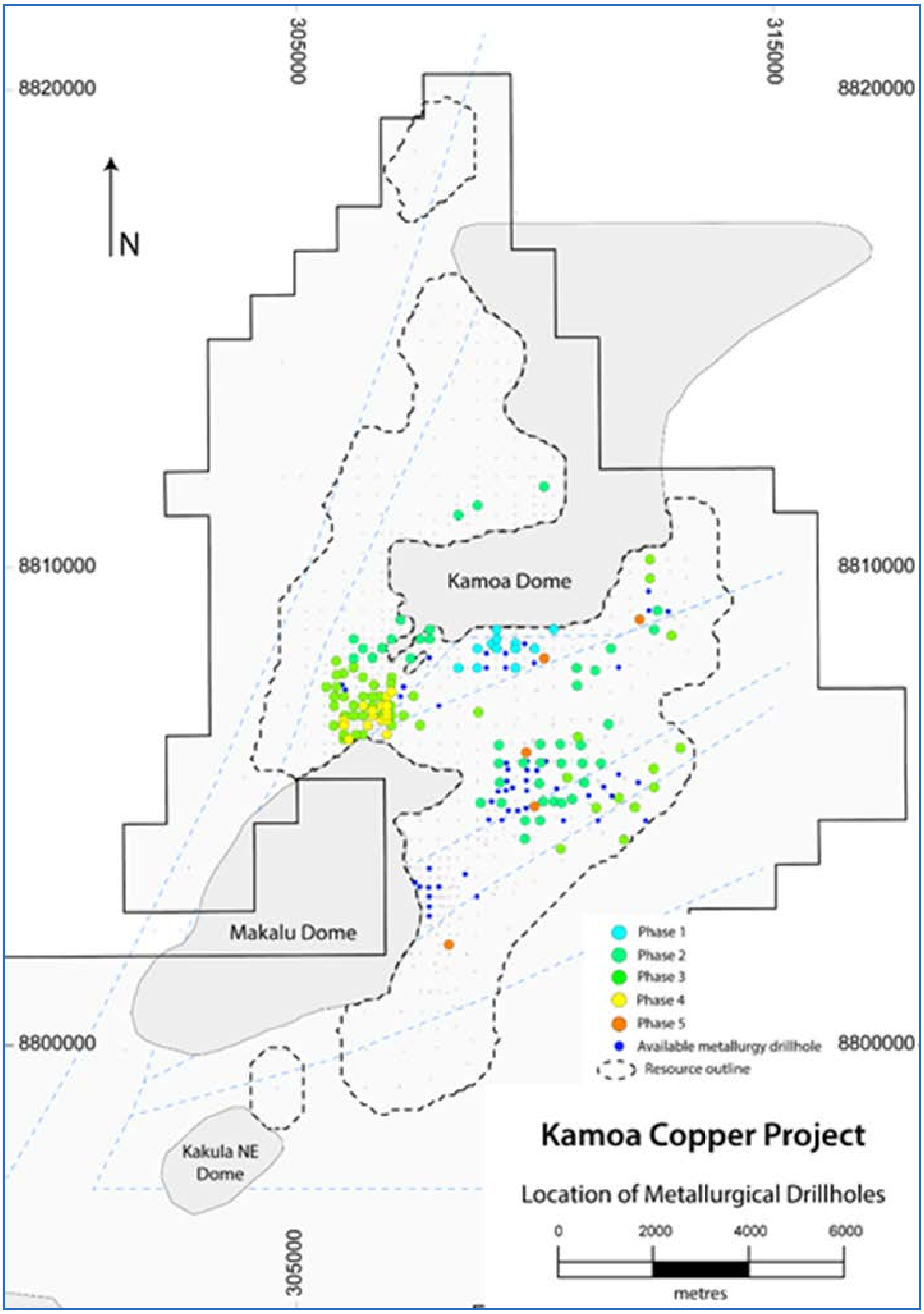


Figure provided by Ivanhoe, 2016.

13.2.3 Kamoa Comminution Testwork

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

Table 13.2 Comminution Programme, Sample Numbers Tested

<table>
<tr><th></th><th colspan="2">Bench Scale Comminution Testwork</th><th>Phase 1</th><th>Phase 2</th><th>Phase 4</th><th>Phase 5</th></tr>
<tr><td>1</td><td colspan="2">SMC test</td><td>3 samples</td><td>8 samples</td><td>6 samples</td><td>6 samples</td></tr>
<tr><td>2</td><td colspan="2">BRWI at 1180 µm</td><td>3 samples</td><td>6 samples</td><td>1 sample</td><td>6 samples</td></tr>
<tr><td rowspan="4">3</td><td rowspan="4">BBWI</td><td>at 212 µm</td><td>–</td><td>–</td><td>–</td><td>1 sample</td></tr>
<tr><td>at 106 µm</td><td>3 samples</td><td>8 samples</td><td>6 samples</td><td>6 samples</td></tr>
<tr><td>at 75 µm</td><td>3 samples</td><td>–</td><td>–</td><td>–</td></tr>
<tr><td>at 53 µm</td><td>–</td><td>–</td><td>6 samples</td><td>6 samples</td></tr>
<tr><td>4</td><td colspan="2">Ai</td><td>1 sample</td><td>8 samples</td><td>6 samples</td><td>6 samples</td></tr>
<tr><td>5</td><td colspan="2">CWI</td><td>–</td><td>–</td><td>6 samples</td><td>6 samples</td></tr>
</table>

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, hanging wall)	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 30–40 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 (hanging wall, 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 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 (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, hanging wall)			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 hanging wall 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 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 (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, hanging wall)	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 17–20 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 8 mm. As BRWI values up to 24 kWh/t were obtained, consideration should be given to generating even finer mill feed (a top-size of eight or even 7 mm) in the feed crushing stage.

13.2.3.4 Crushability Summary

The Bond Crushing Work Index test (CWI) measures how difficult particles in the 50–75 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 (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, hanging wall)	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 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, hanging wall)	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 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, hanging wall)	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 an 8 mm ball mill feed top-size is preferred.

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 Kamoa 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 sulfide 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 sulfide 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 sulfide population locked in silicates at sulfide 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.
- A MF2 rougher flotation scheme achieved slightly higher recoveries than a typical mill float (MF1) arrangement.
- Cleaning of concentrates after dual regrinding to 20–30 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

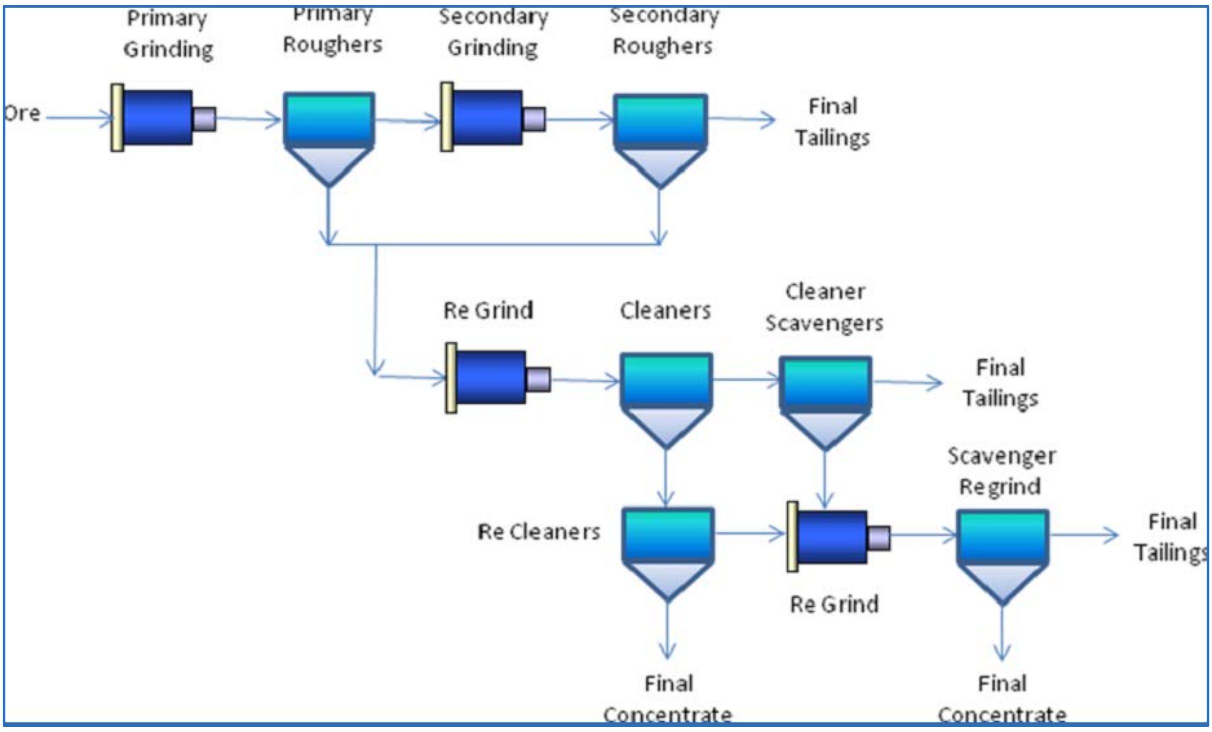


Image courtesy of Mintek, 2010.

13.2.4.2 Phase 2 (2010–2011) 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 sulfide stoichiometry), similar overall copper recoveries were indicated.

The 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 (2011–2013) – Xstrata Process Support (XPS) Laboratories in Canada

Flotation testing for Phase 2 and Phase 3 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" (Figure 13.3) was developed in Phase 2 that was tailored to selective recovery of the finer grained sulfide 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

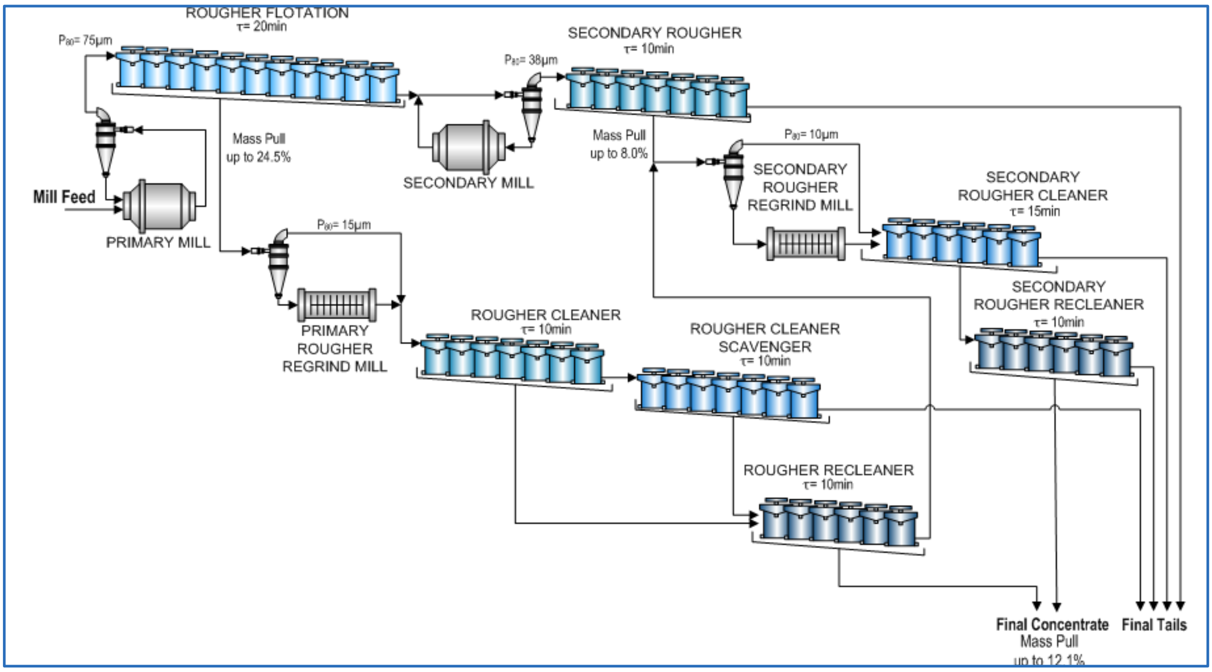


Figure by Hatch, 2013.

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

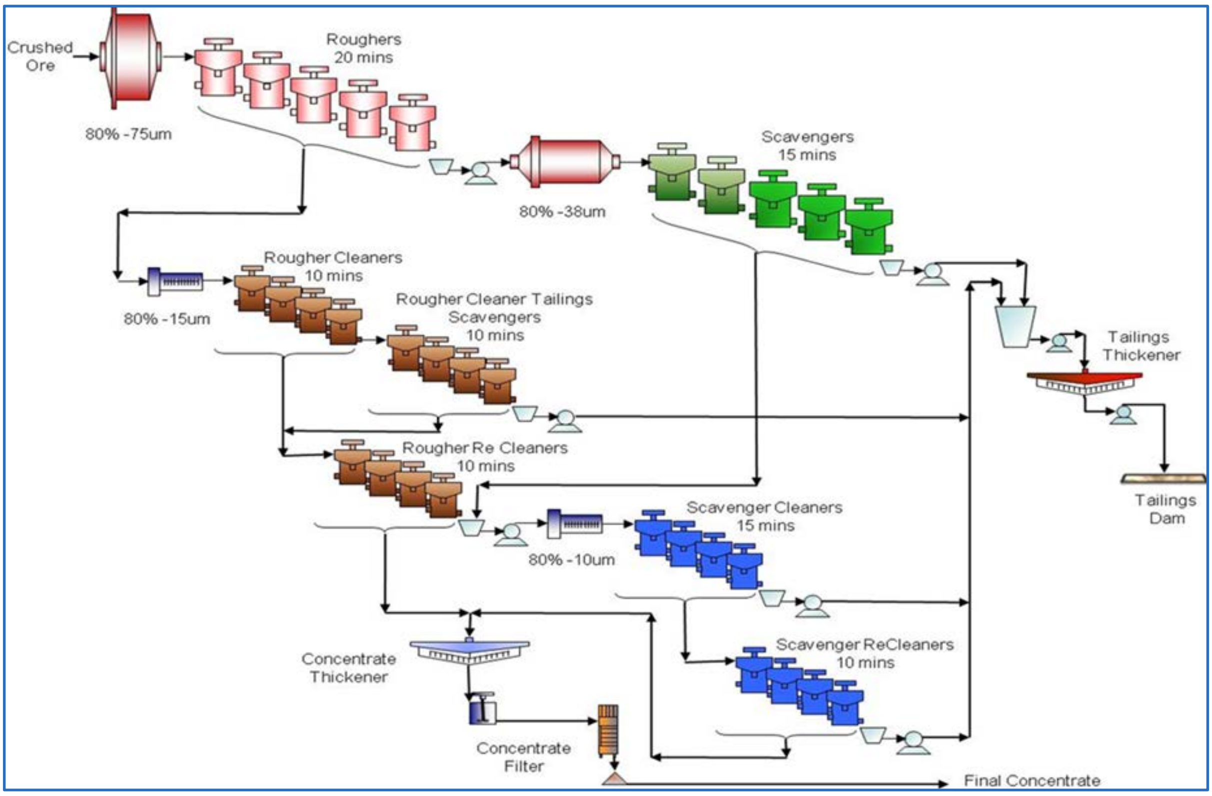


Image courtesy XPS, 2013.

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 sulfide component with more than 40% of the copper sulfide minerals having a grain size of less than 10 µm. Evidence of fine locked sulfides 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 sulfide 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-sulfide sulfur minerals were identified in the Supergene or Hypogene samples such that total sulfur analysis could reasonably be assumed to be equivalent to the sulfide sulfur analysis.
- Other than silica, there are no penalty elements present that reach problematic levels in the concentrate.
- Hanging wall 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 (80–87%) at concentrate grades of between 18–25% 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	38.70	–7.2
	% Cu	9.94	10.00	0.6
	Rec Cu	94.5	93.90	–0.6
Final Concentrate	% Mass	15.1	13.20	–12.6
	% Cu	26.3	27.60	4.9
	Rec Cu	90.8	88.20	–2.9
Tail	% Mass	84.9	86.80	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 hanging wall composites, footwall composites and variability samples from what has been termed the Minzone. Minzone refers to the single 6–12 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

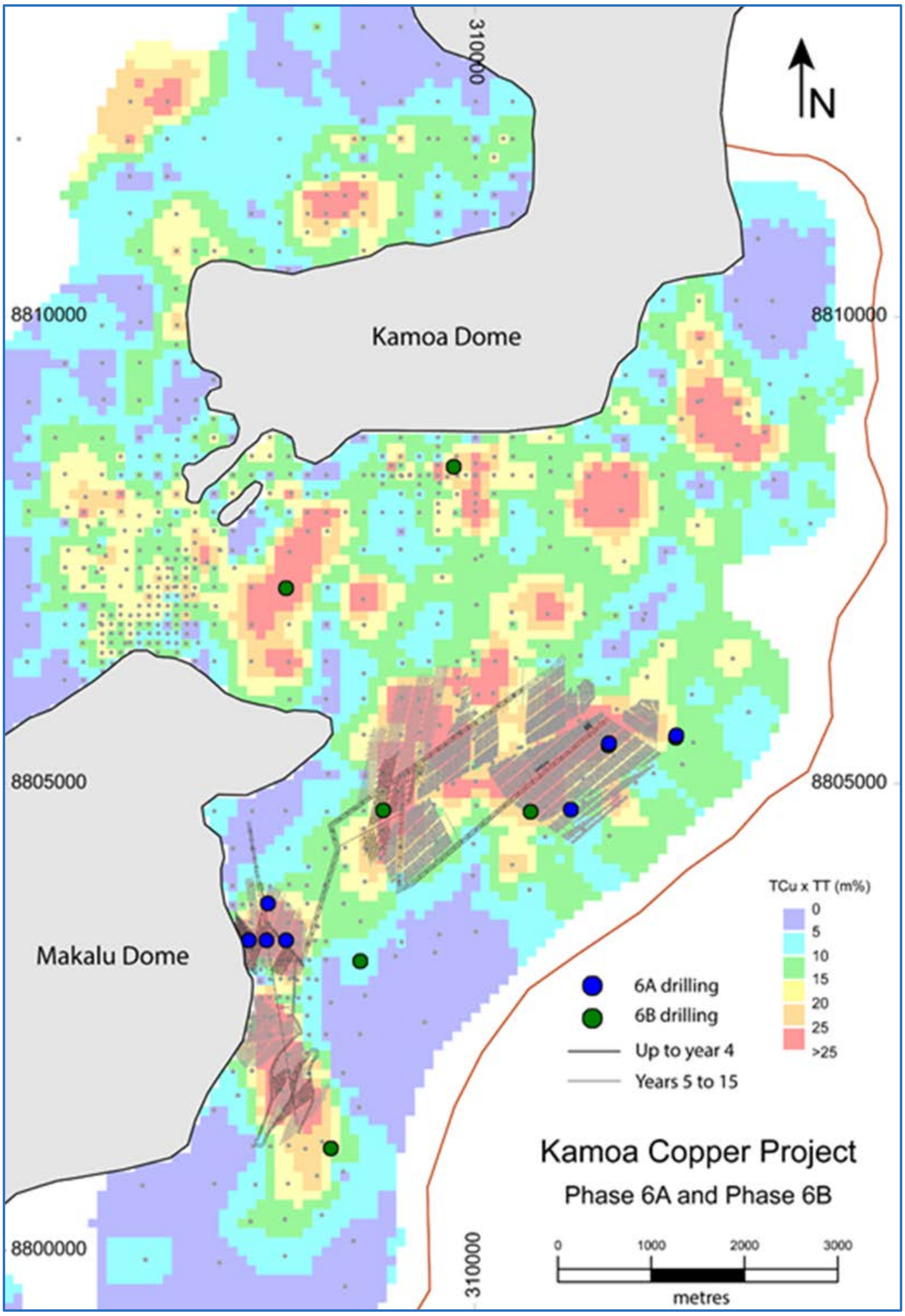


Figure provided by Ivanhoe, 2016.

Table 13.10 Phase 6 Comminution Summary

Sample ID	SG	BRWi	BBWi (kWh/t)		UCS (Mpa)	CWI (kWh/t)	Ai	A*b
		kWh/t	53 µm	106 µm	Avg.	Avg.	g	
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 hanging wall 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	–	17–38	–
	BBWI (106 µm)	17.7	13–22	Yes
	BRWI	21.5	16–23	Yes
	Ai	0.060	0.04–0.27	Yes
	CWI	9.7	9–20	No
	UCS	119	95–255	Yes
Siltstone (Hanging wall)	Axb	–	21–29	Yes
	BBWI (106 µm)	15.7	16–20	Yes
	BRWI	11.8	20–24	No
	Ai	0.069	0.04–0.05	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 7.2–10.9 kWh/t. In contrast the four Phase 5 Minzone samples vary from 9–20 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) used 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

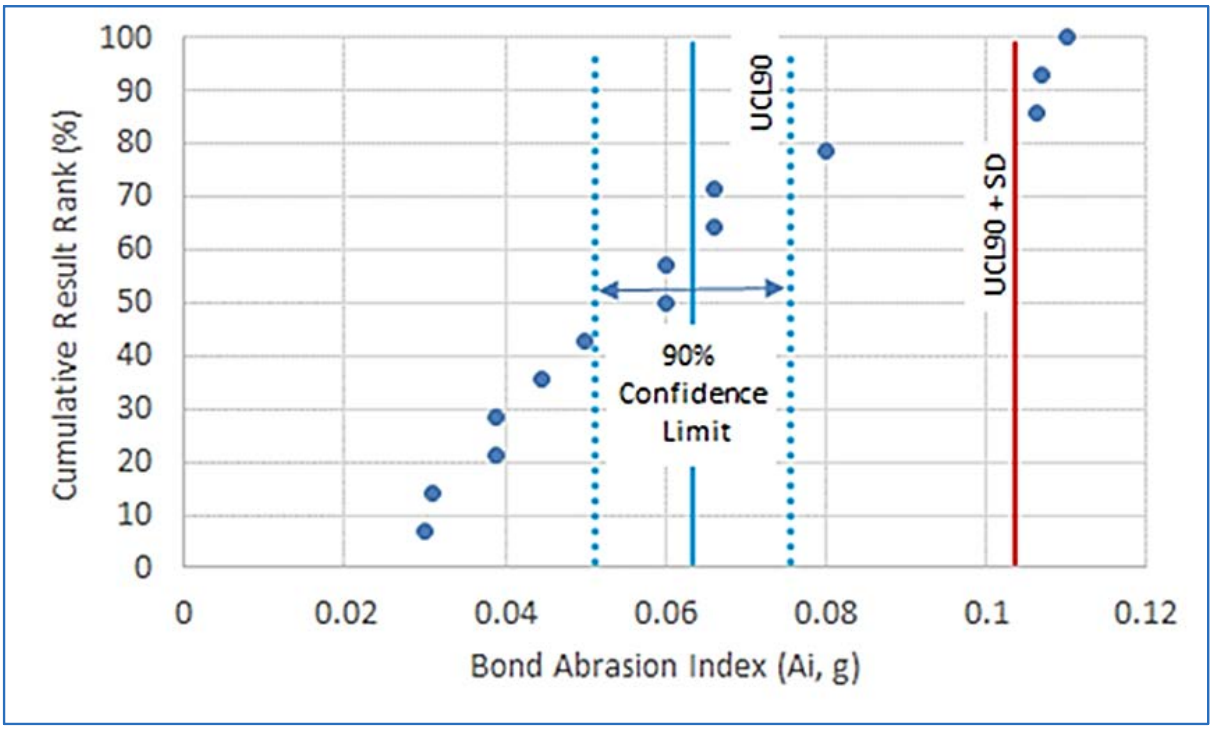


Image courtesy of Amec Foster Wheeler, 2016.

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

Figure 13.7 Drill Collars for Phase 6 Flotation Test Composite Samples

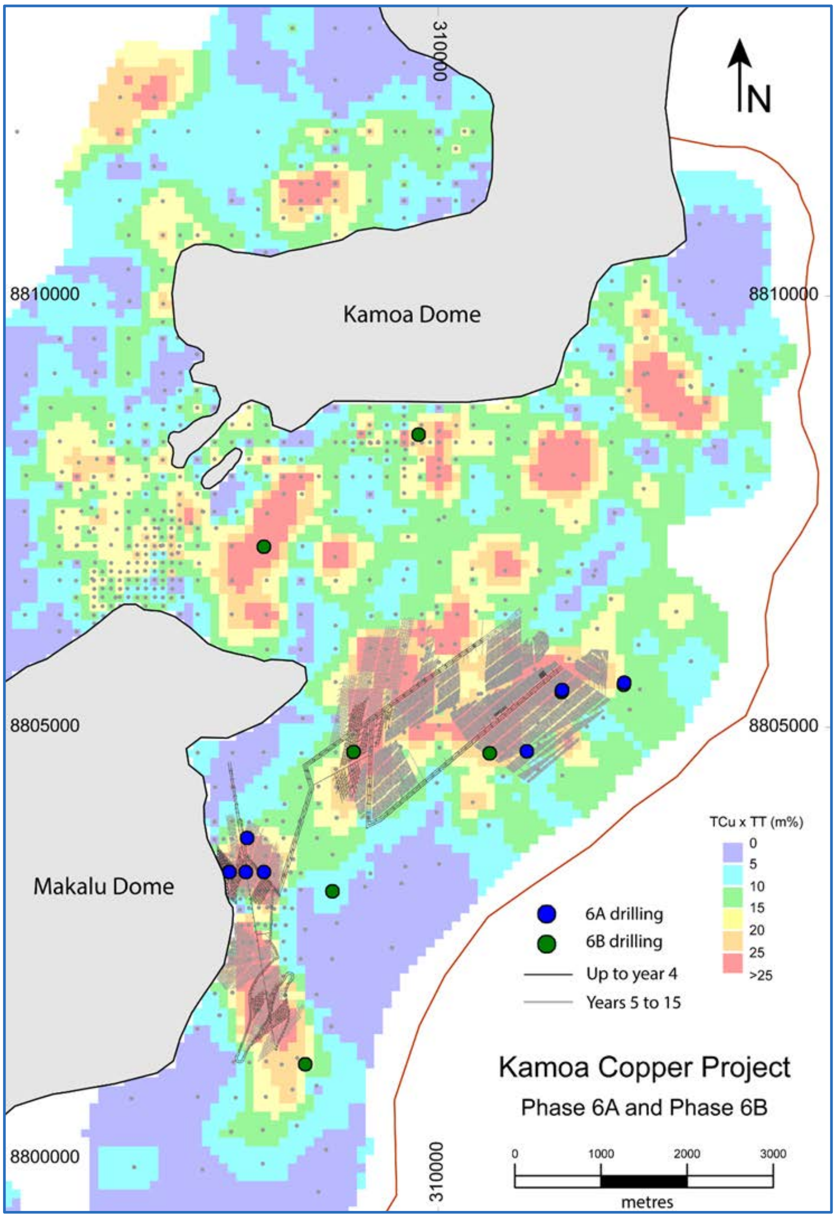


Figure provided by Ivanhoe, 2016.

The Phase 6 samples were prepared in sets containing a development composite (DC) and two individual composites based on copper sulfide 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 (%)	Al_2O_3 (%)	MgO (%)	SiO_2 (%)
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 sulfur as shown in Figure 13.8.

Figure 13.8 Copper to Sulfur Ratios in Phase 6 Composites

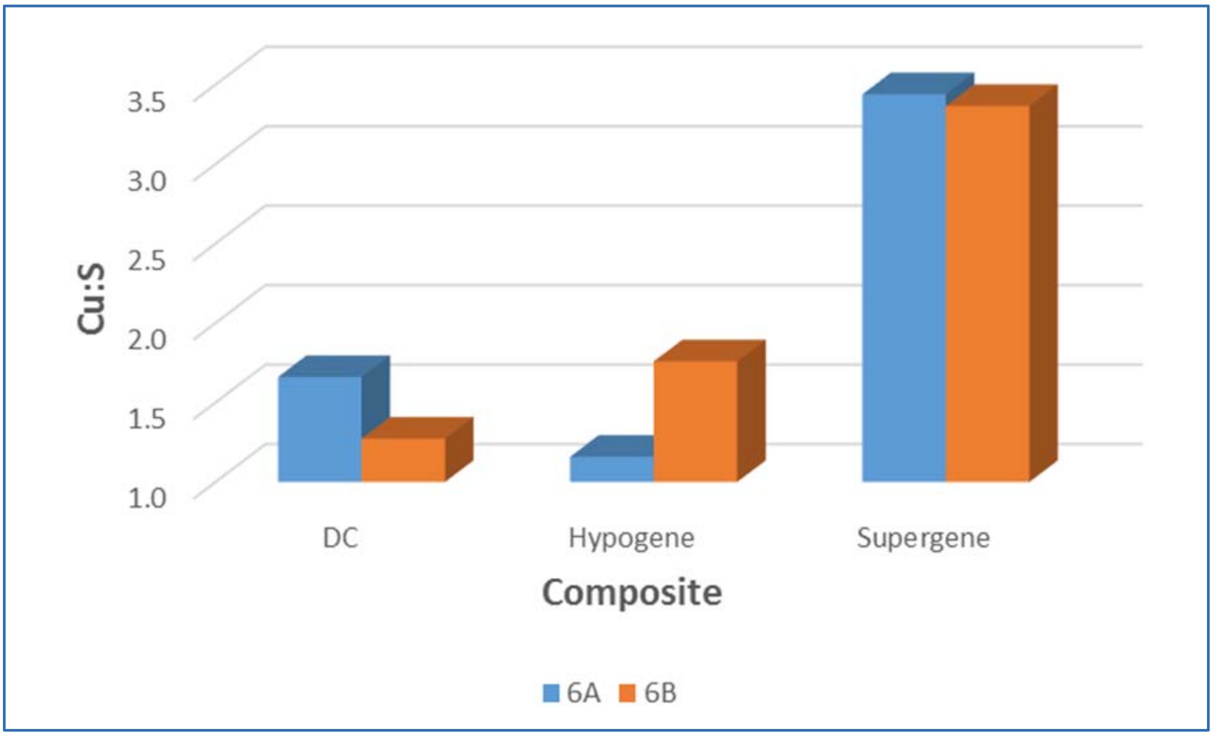


Image courtesy of Amec Foster Wheeler, 2016.

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

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

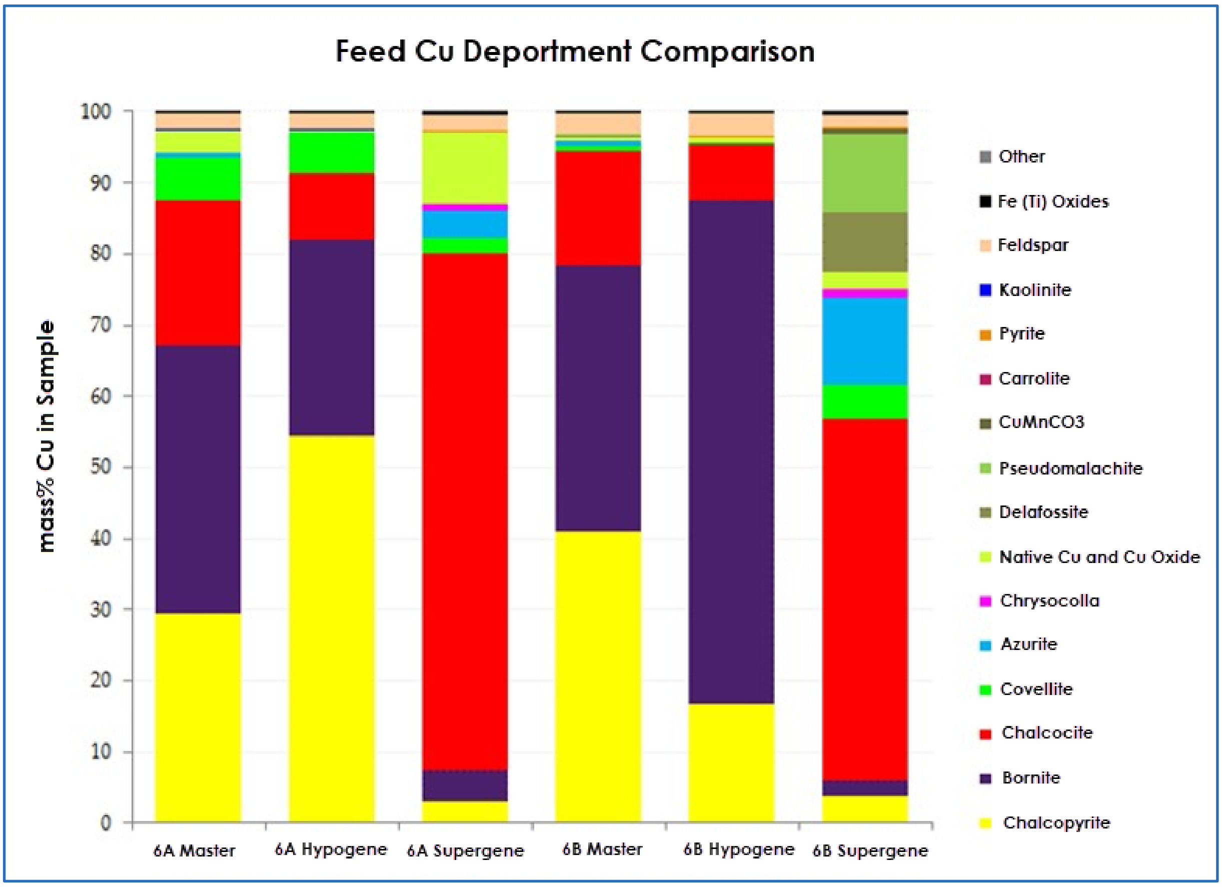


Image courtesy XPS, 2015.

Master sample is an alternative name for the 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

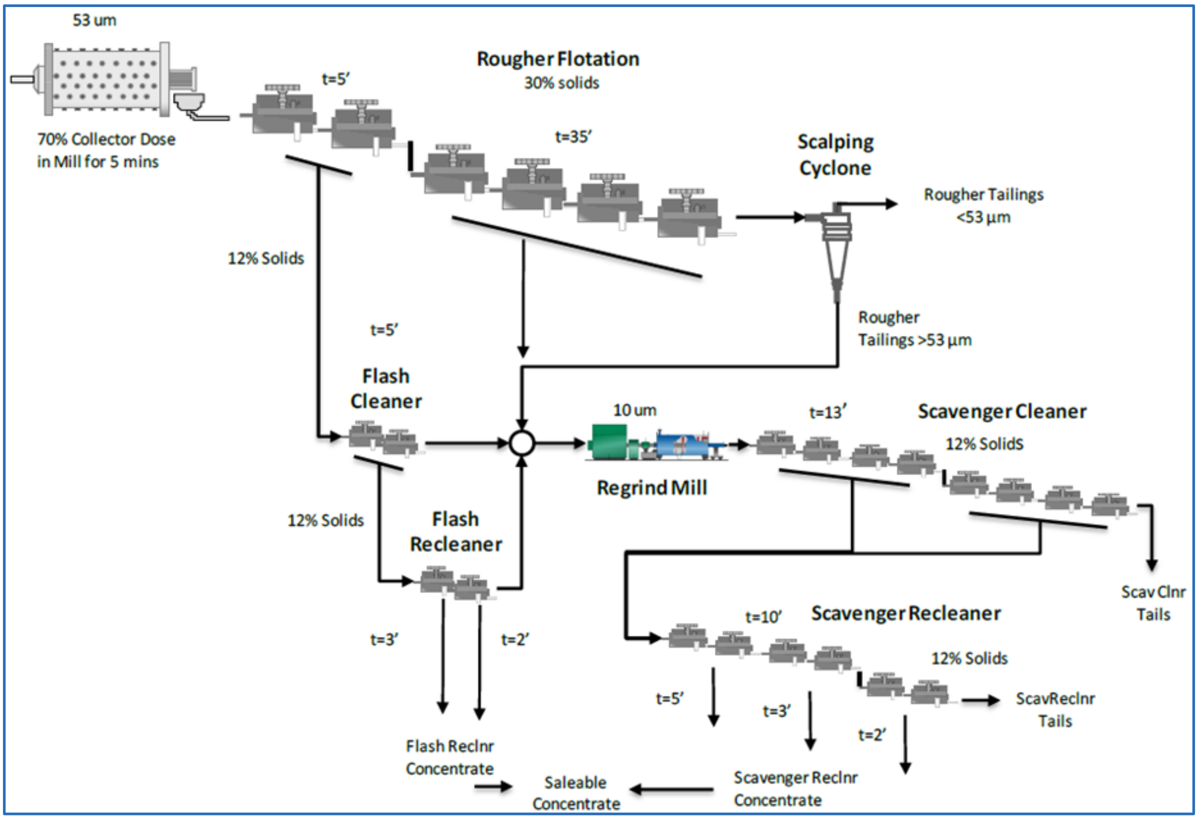


Image courtesy XPS, 2015.

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	14.60	16.30	0.48	3.76
	90:10 H: S	8.75	37.2	88.7	6.13	22.90	0.45	3.58
	Hypo	8.98	35.7	90.0	4.92	23.40	0.40	3.56
	Super	5.62	48.5	75.3	14.50	8.47	0.95	3.62
6B	DC	8.14	37.0	92.3	7.62	22.70	0.28	3.26
	Hypo	6.29	44.5	91.9	10.60	15.40	0.26	3.05
	Super	5.96	46.5	69.4	15.80	10.60	1.30	3.99
15-year Comp		7.34	39.0	88.1	11.00	17.80	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:

- 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.
- Scalping using cyclones would also result in a loss of some of the oversize to overflow due to inefficiency.
- An alternative to cyclone scalping of the tailings would be to grind finer before the roughers.

- 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.
- 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 PS

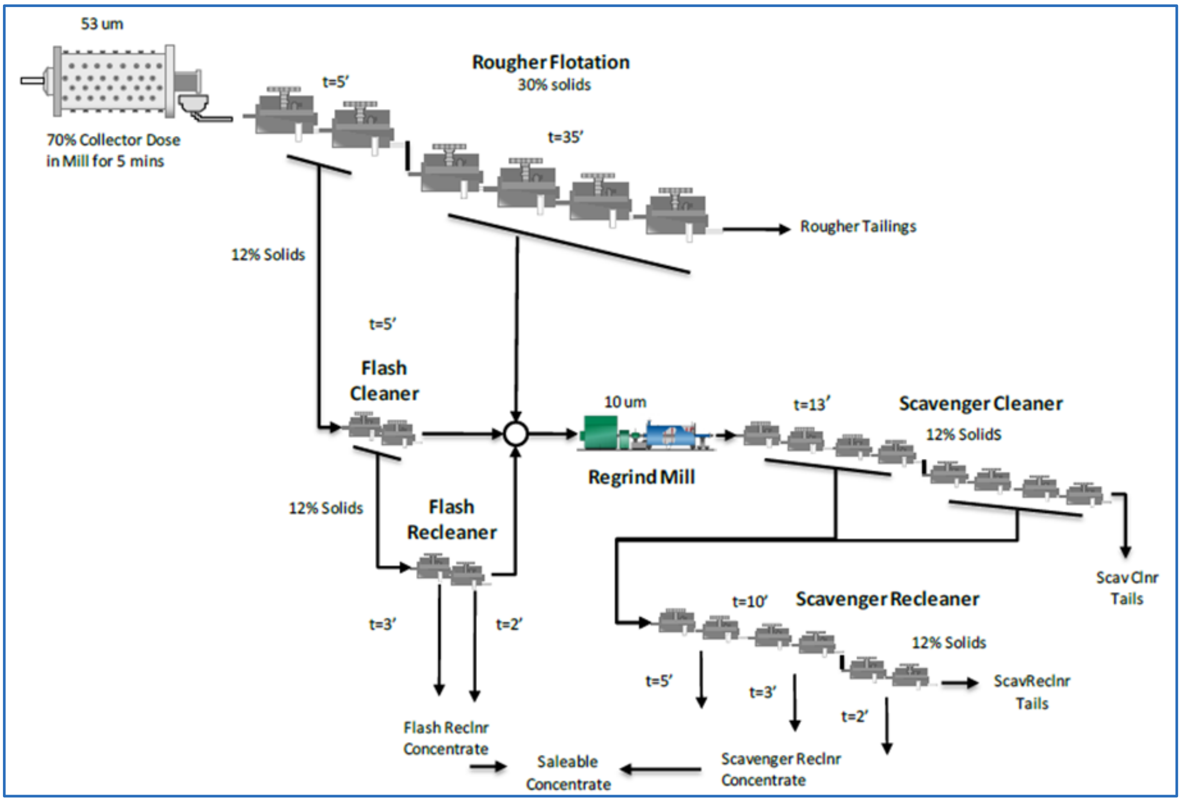


Image courtesy XPS, 2015.

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	11.10	16.80	0.56	3.74
	90:10 H: S	8.33	37.0	85.4	6.34	22.00	0.58	3.61
	Hypogene	8.48	36.0	86.1	4.00	21.00	0.54	3.54
	Supergene	5.25	53.5	72.3	13.50	13.40	1.14	3.89
6B	DC	8.07	35.4	89.2	9.45	21.30	0.37	3.20
	Hypogene	7.17	35.5	86.9	19.20	13.50	0.41	2.93
	Supergene	6.02	41.2	65.3	19.30	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

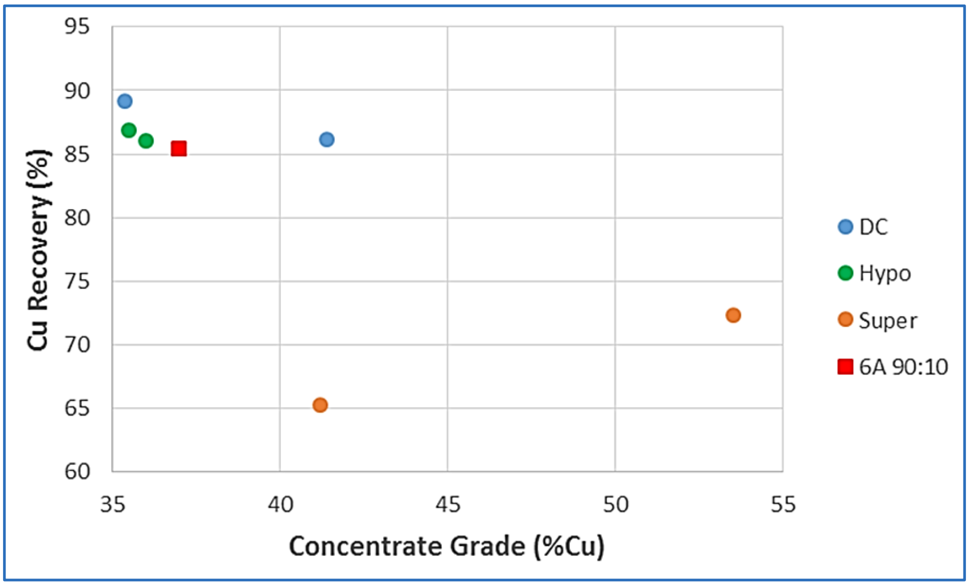


Image courtesy of Amec Foster Wheeler, 2016.

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-sulfide 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)

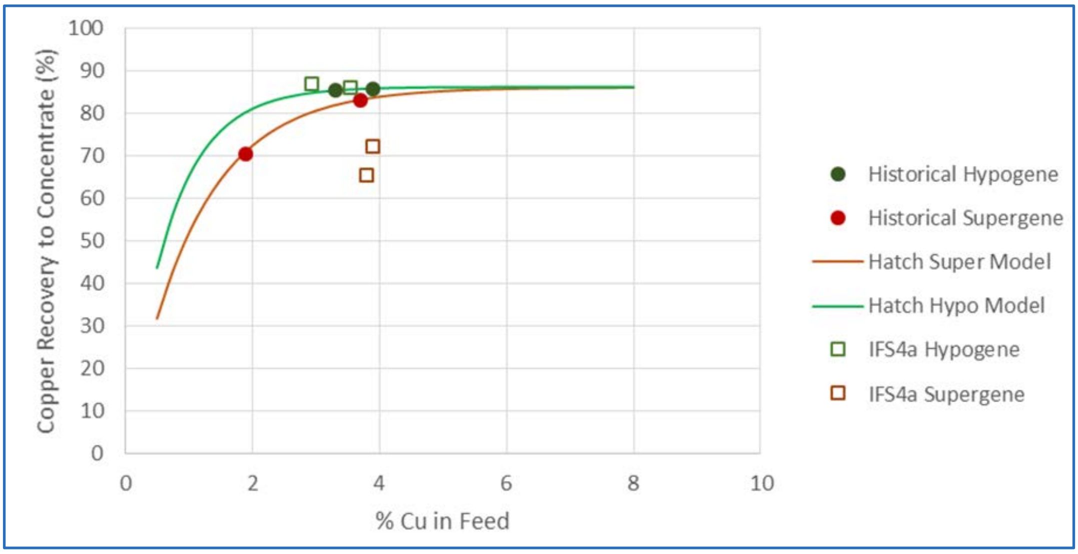


Image courtesy of Amec Foster Wheeler, 2016.

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

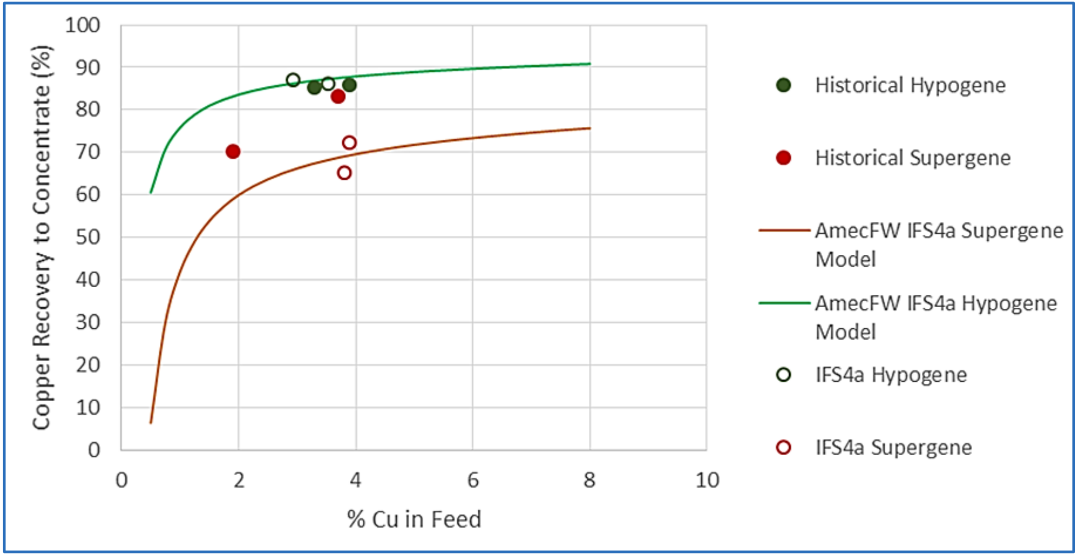


Image courtesy of Amec Foster Wheeler, 2016.

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 5–15% (it is thought that the ultra-fine component of the sulfide 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

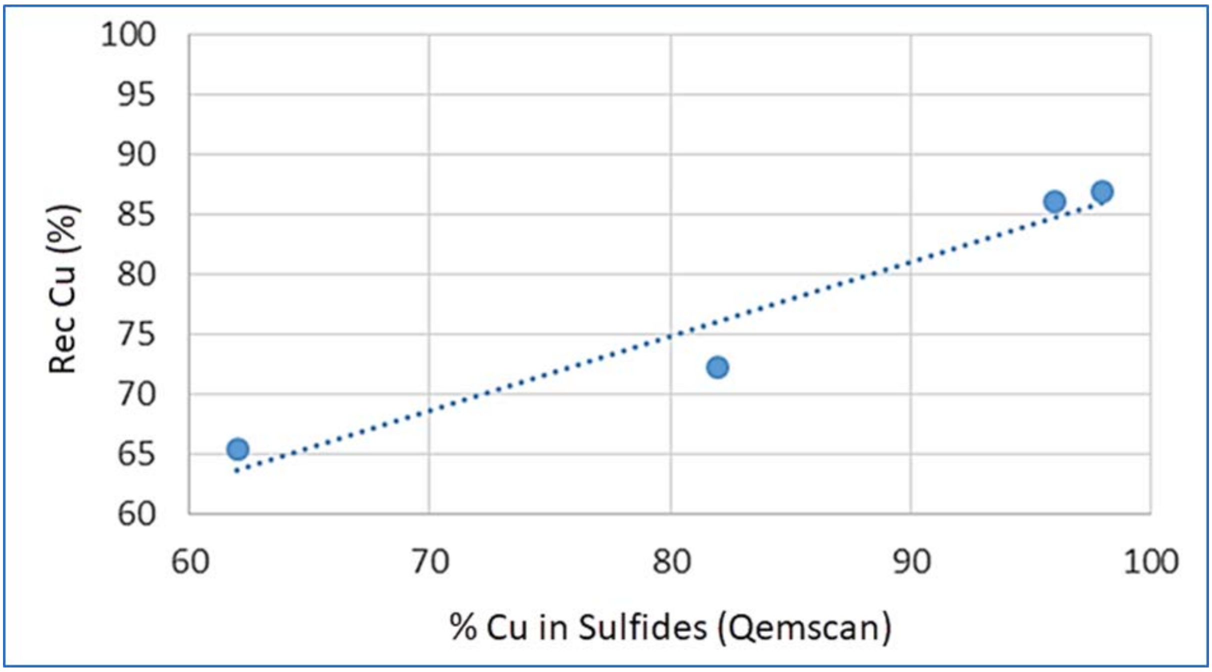


Image courtesy of Amec Foster Wheeler, 2016.

This strong relationship between recoverable copper and copper in sulfides is expected. Almost all oxide copper minerals, together with native copper, are not readily floated in a standard copper sulfide 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

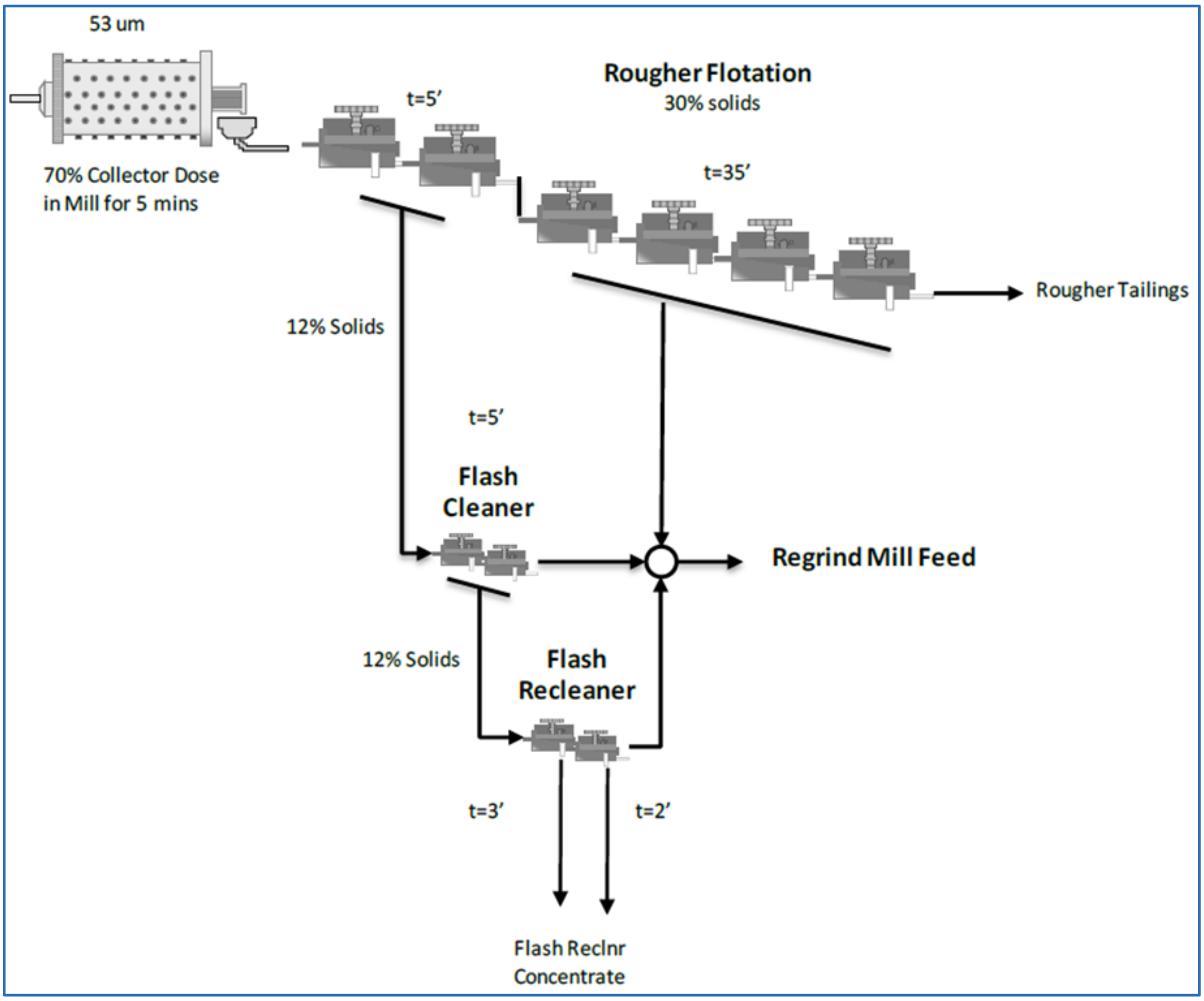


Image courtesy XPS, 2015.

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

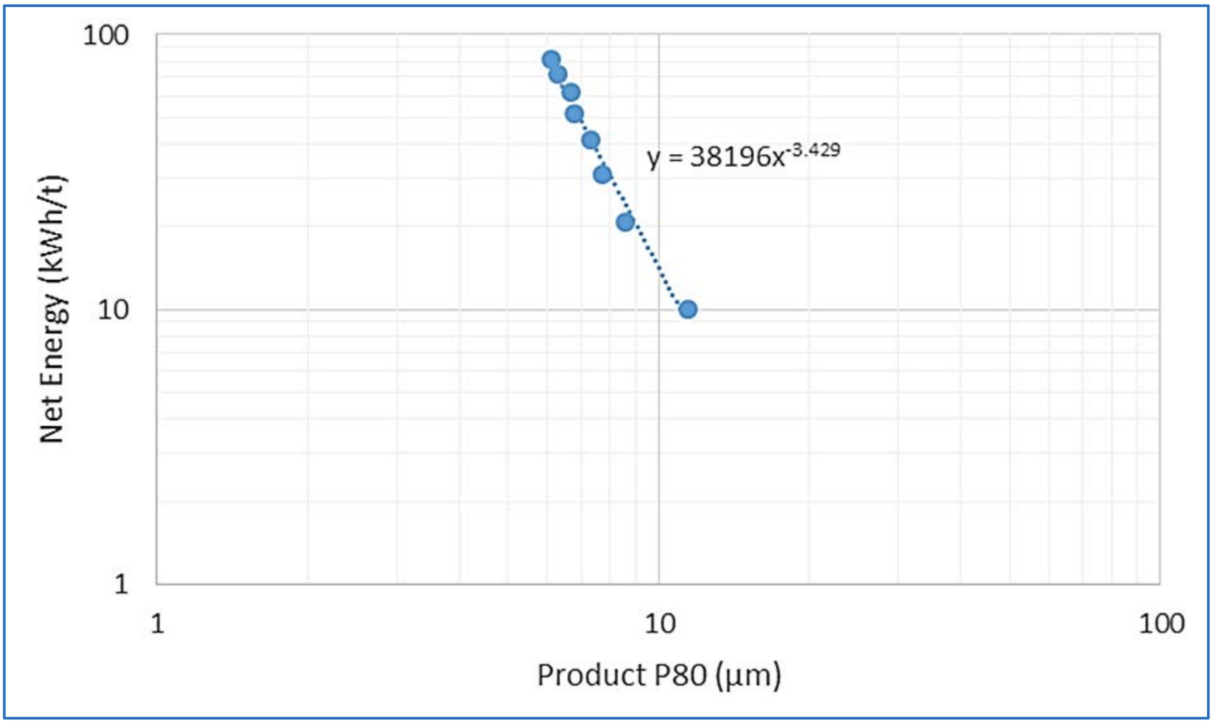


Image courtesy of Amec Foster Wheeler, 2016.

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

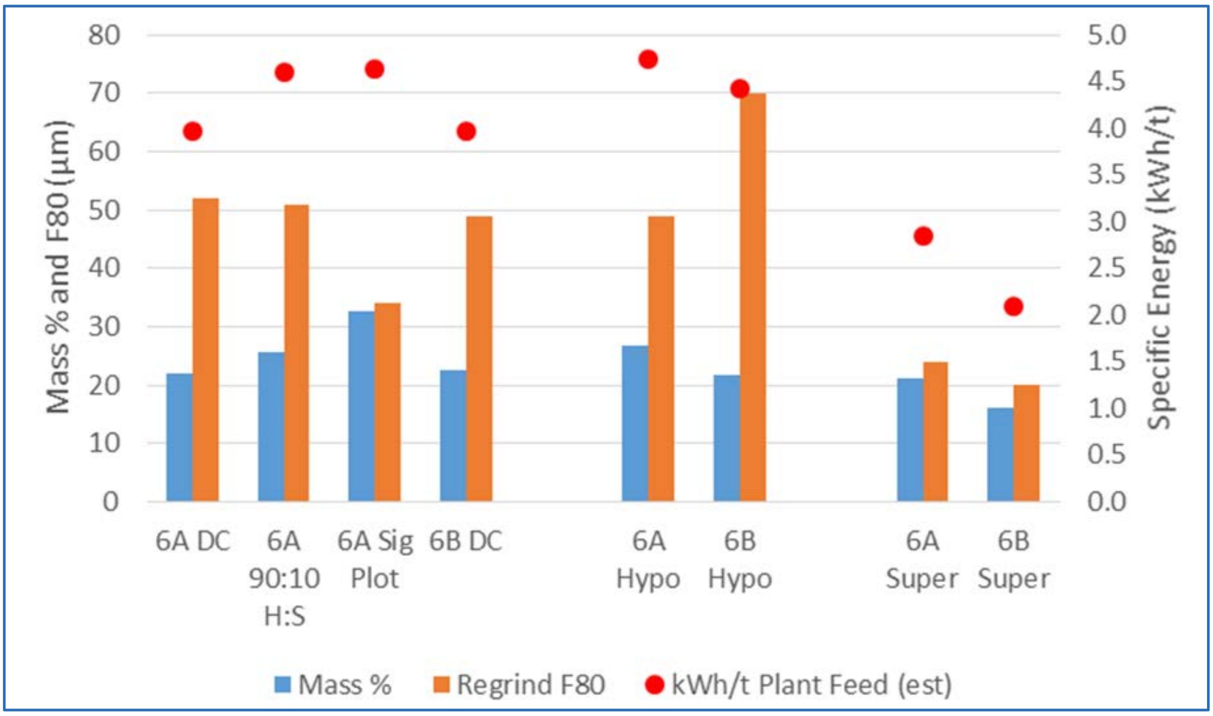


Image courtesy of Amec Foster Wheeler, 2016.

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 Year-15 PFS mining areas.

Figure 13.19 Planned Phase 6 Variability Samples

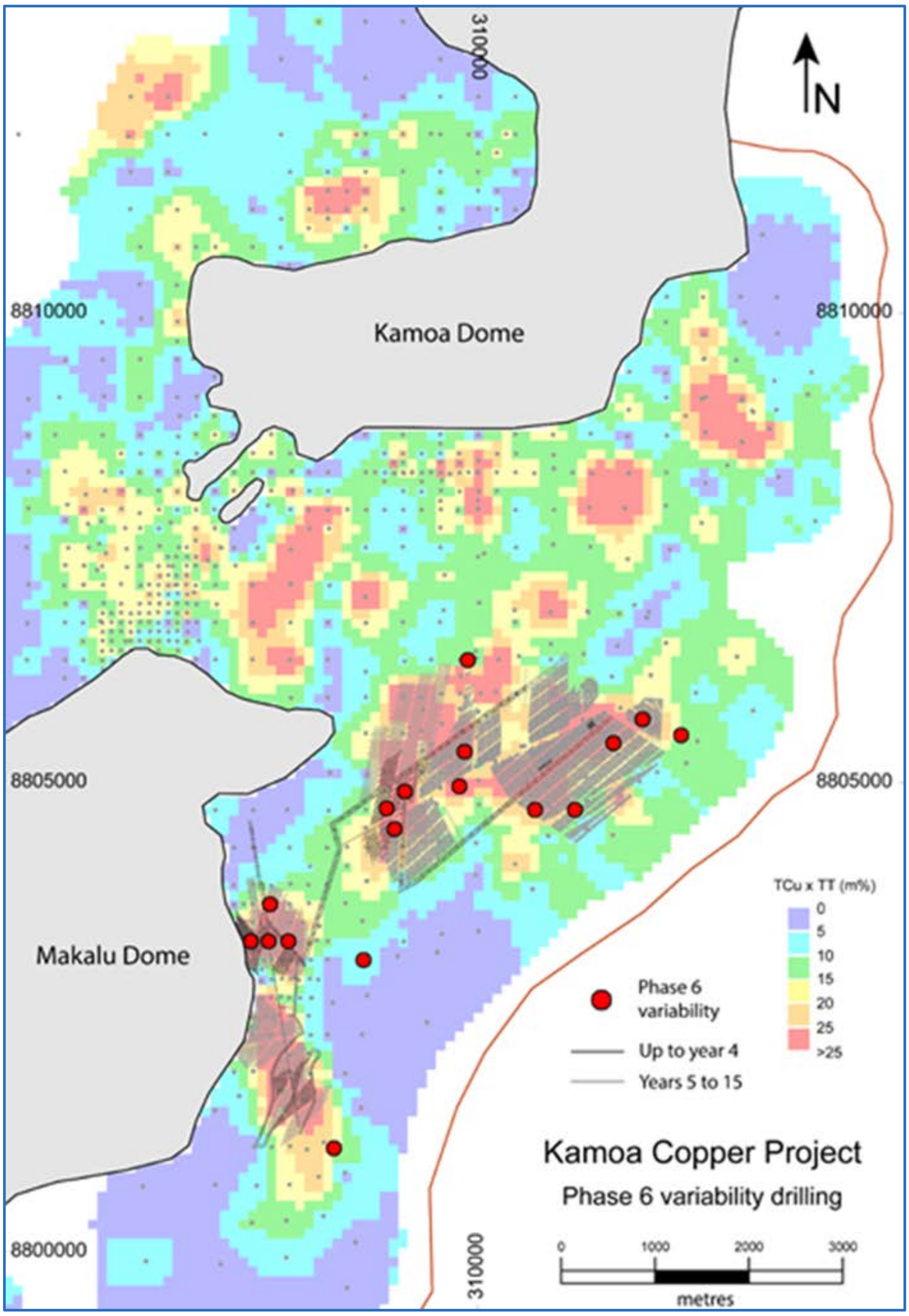


Figure provided by Ivanhoe, 2016.

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 sulfide mineralisation exists in two basic modes regardless of copper sulfide mineral. Coarse copper sulfides, 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 sulfides 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 sulfide deposition phase. In the surrounding rock matrix there are smaller mantled clasts and visible blebs of chalcopyrite (and other sulfides). What cannot be seen in the photograph is the dispersion of 1–10 µm (0.0001–0.001 cm) copper sulfides 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 Sulfide Liberation in Rougher Flotation

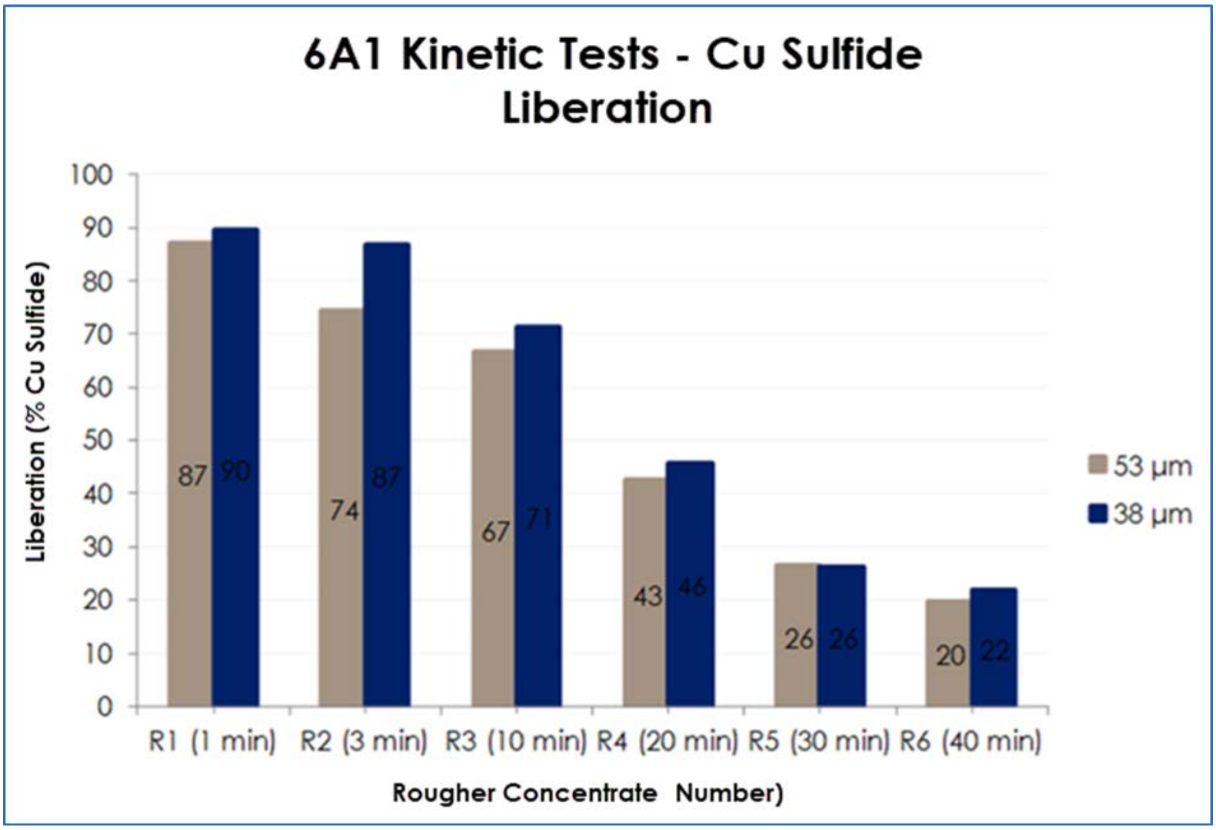


Image courtesy XPS, 2015.

The highly-liberated copper sulfides are floated preferentially while the poorly-liberated sulfides 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 sulfide 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 sulfides 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

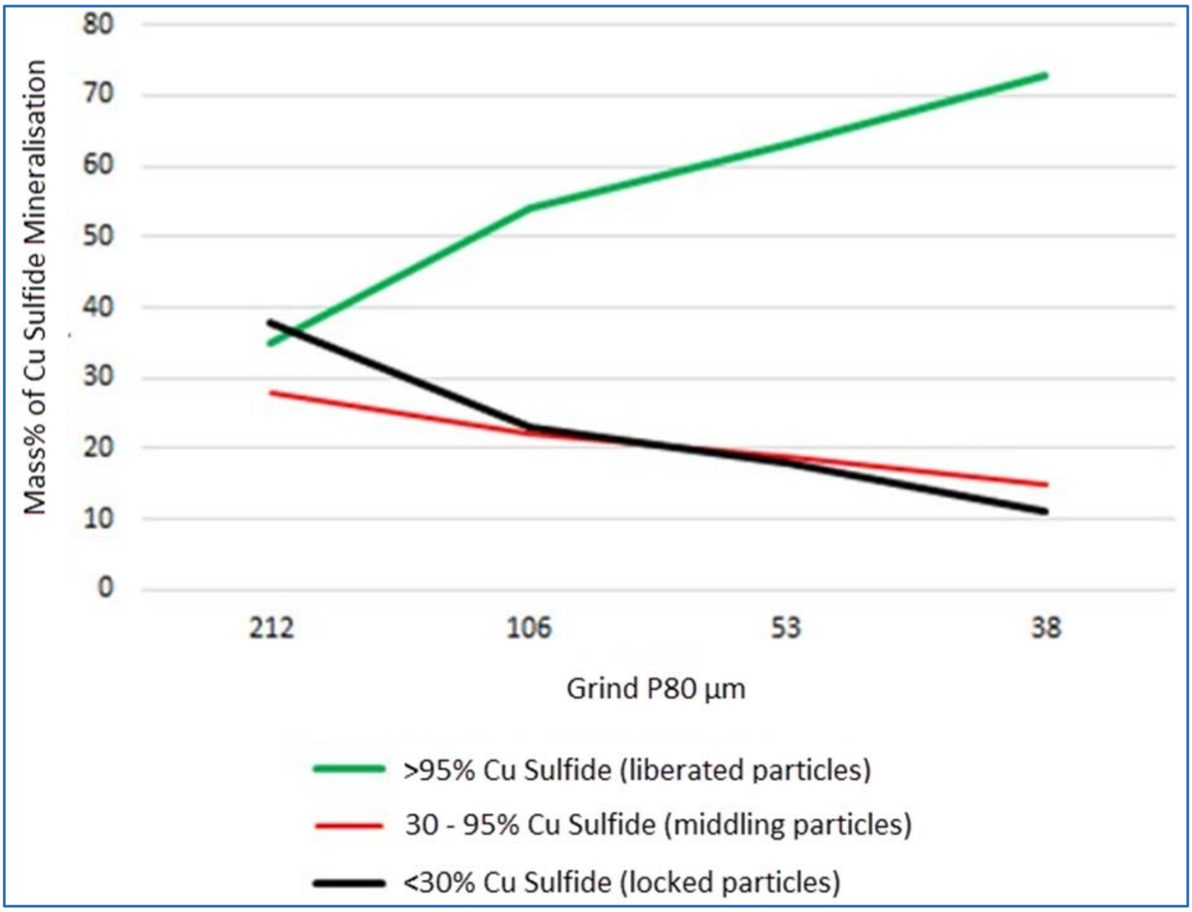


Image courtesy Amec Foster Wheeler, 2018: from XPS data, 2014.

At the fine grind P_{80} of 38 µm, 27% of the copper sulfides 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 Sulfides Liberation Map – Rougher Concentrates R3–R6

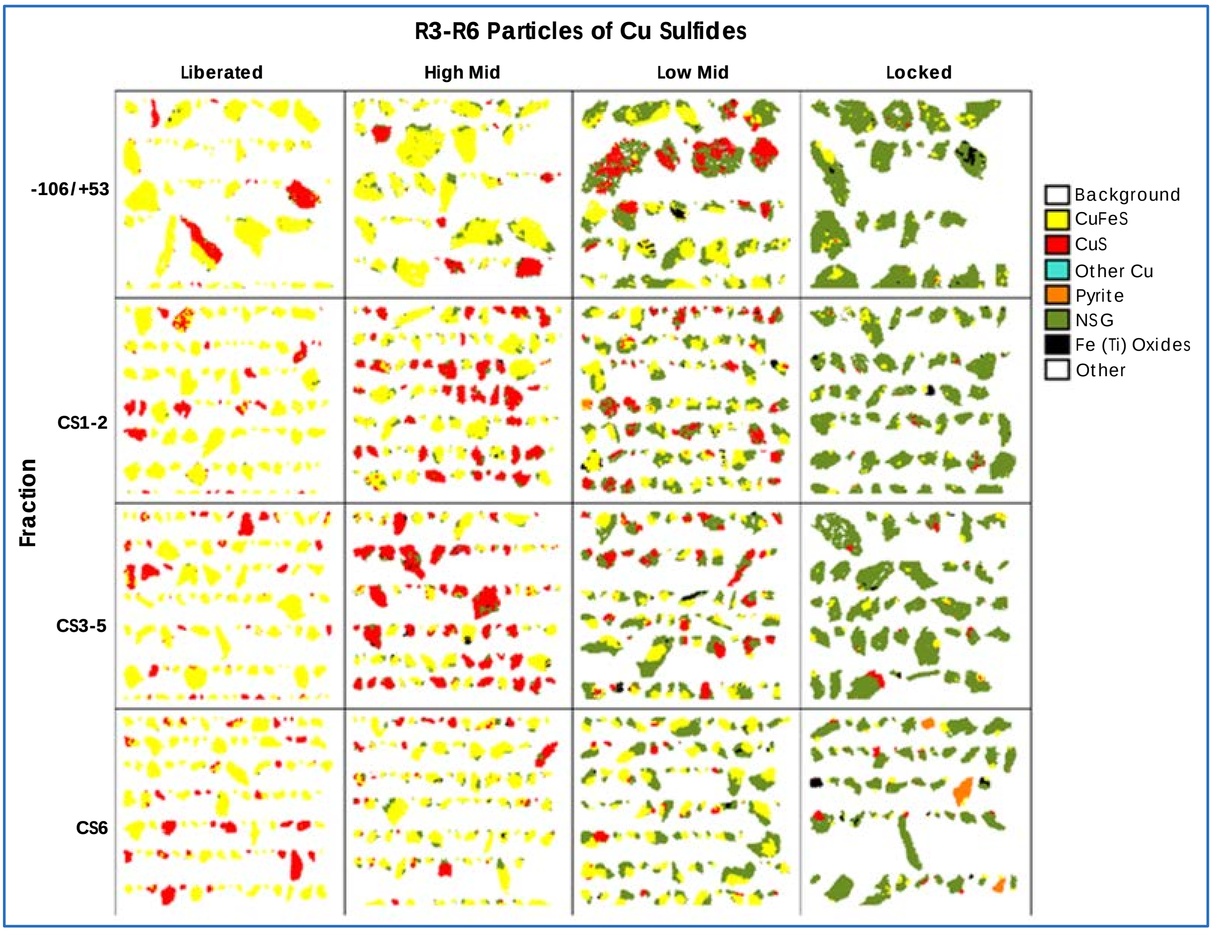


Image courtesy XPS, 2015.

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 sulfide phases in the CS6 particles are typically 1–3 µm. This poor liberation of fine sulfides 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 sulfide at the surface and the "low Mid" and "Locked" particles in the image shows that this is generally the case.

The pervasive fine copper sulfides cause large amounts of attached silicates to be recovered in rougher flotation and this leads to the high rougher mass pull values (20–40%) typical in the test programs. At coarse grinds, such as 150 µm P_{80}, large silicate particles invariably have exposed fine copper sulfides on the surface and are able to float.

The fine sulfides 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 sulfides 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 Sulfides Liberation Map – Rougher Tails

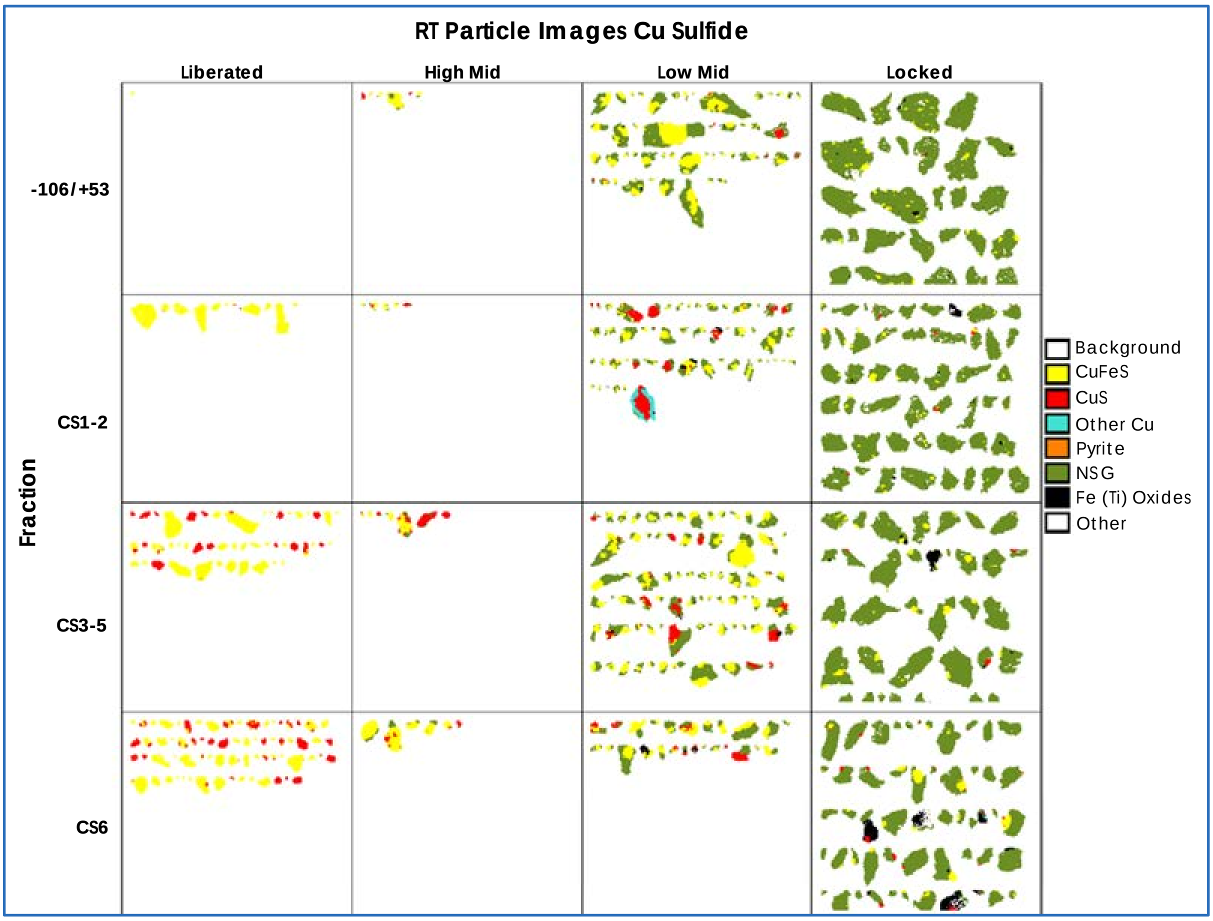


Image courtesy of XPS, 2015.

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 sulfides is passivated or the actual amount of sulfide 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 sulfides are in phases that have average grain sizes of less than 10 µm.

Figure 13.25 Copper Sulfide Phase Size in Rougher Tailings

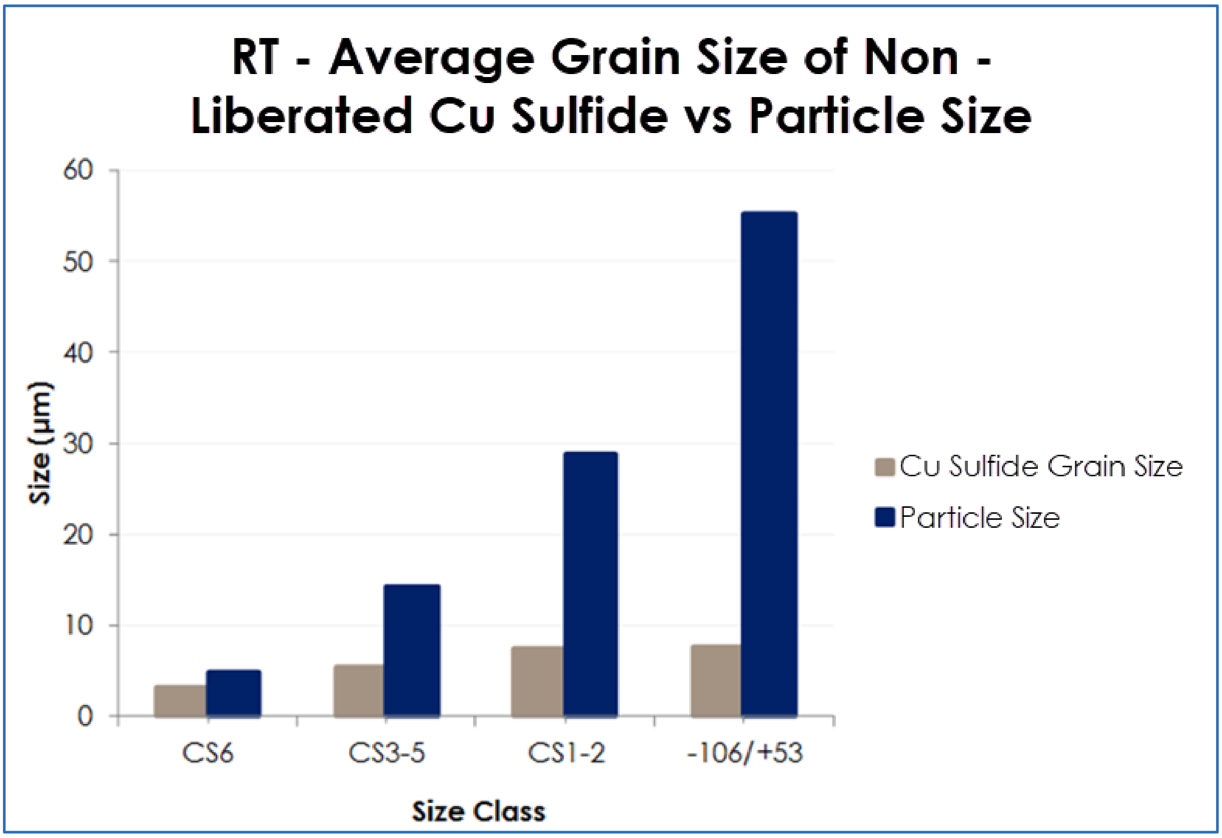


Image courtesy of XPS, 2015.

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 sulfides. 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 Metallurgical 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.

Further testwork was initiated in March 2019 as part of the feasibility study and consisted of:

- A mini-pilot plant campaign including Jameson Cell testwork, conducted by XPS.
- Desliming cyclone testwork, conducted by Multotec, South Africa.
- Flocculant screening testwork, conducted by ChemQuest, South Africa.
- Various slimes and full tailings settling testwork, conducted by Outotec, Paterson & Cooke, and Andritz.
- Concentrate regrind hydro cyclone and signature plot testwork, conducted by Grinding Solutions.
- Flotation tests utilising underground mine water, conducted by XPS.

13.3.1 Kakula Metallurgical Sample Locations and 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.

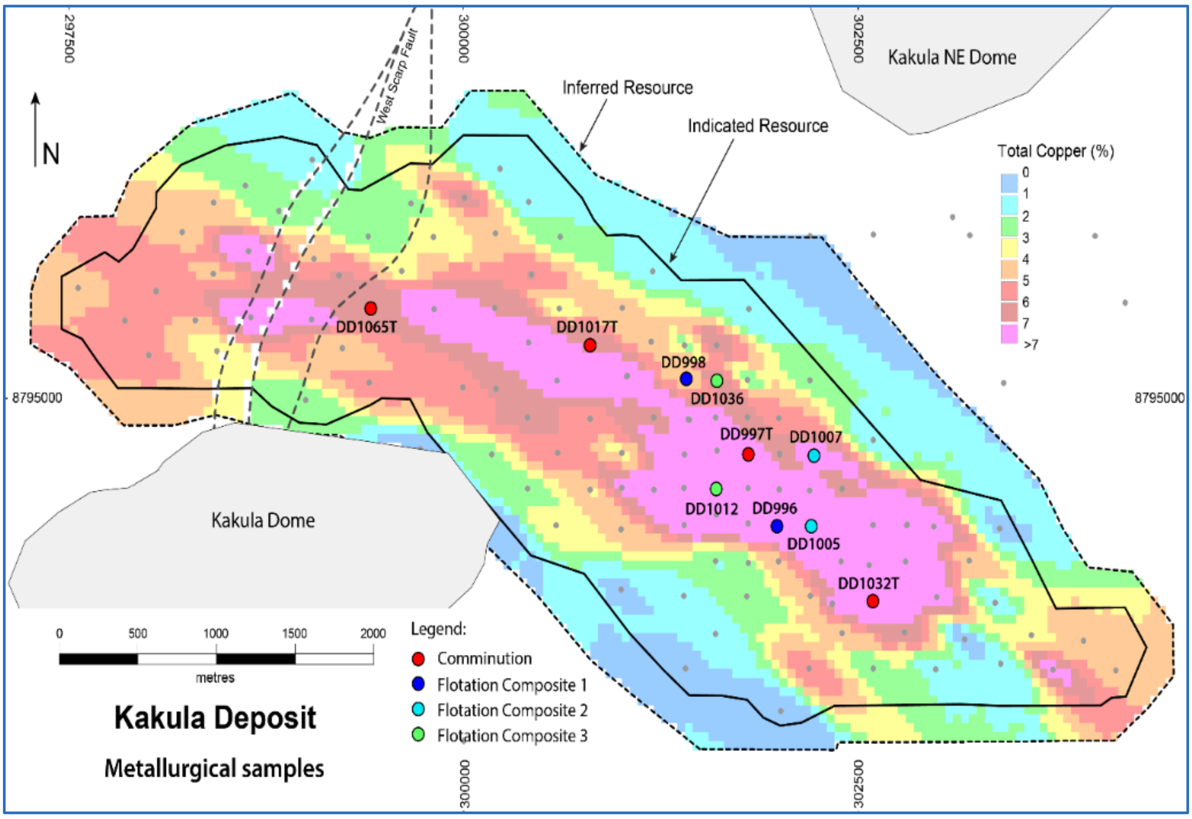
Following successful testing of these early holes, and due to high grade intercepts consistently achieved at Kakula, additional samples from drillholes DD1005 and DD1007 (Flotation Composite 2) were sent to Zijin laboratories, and DD1012 and DD1036 (Flotation Composite 3) were shipped to XPS to verify 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	54.50	4.61	12.10	1.86	4.43
DD998	3.96	1.15	52.70	5.16	10.80	2.62	4.98
Flotation composite 1 (50% DD996: 50% DD998)	4.08	1.20	55.50	5.07	12.70	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 FFS 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 10 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:

- Five individual siltstone samples.
- Two individual diamictite samples.
- One sandstone footwall composite sample.
- One 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 was 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 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 sulfide 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 sulfide grain size distributions between Kamoa 6A1DC and Kakula FC3 samples.

Figure 13.27 Kamoa 6A1DC, Kakula FC3, and Kakula PFS Sample Mineralogy

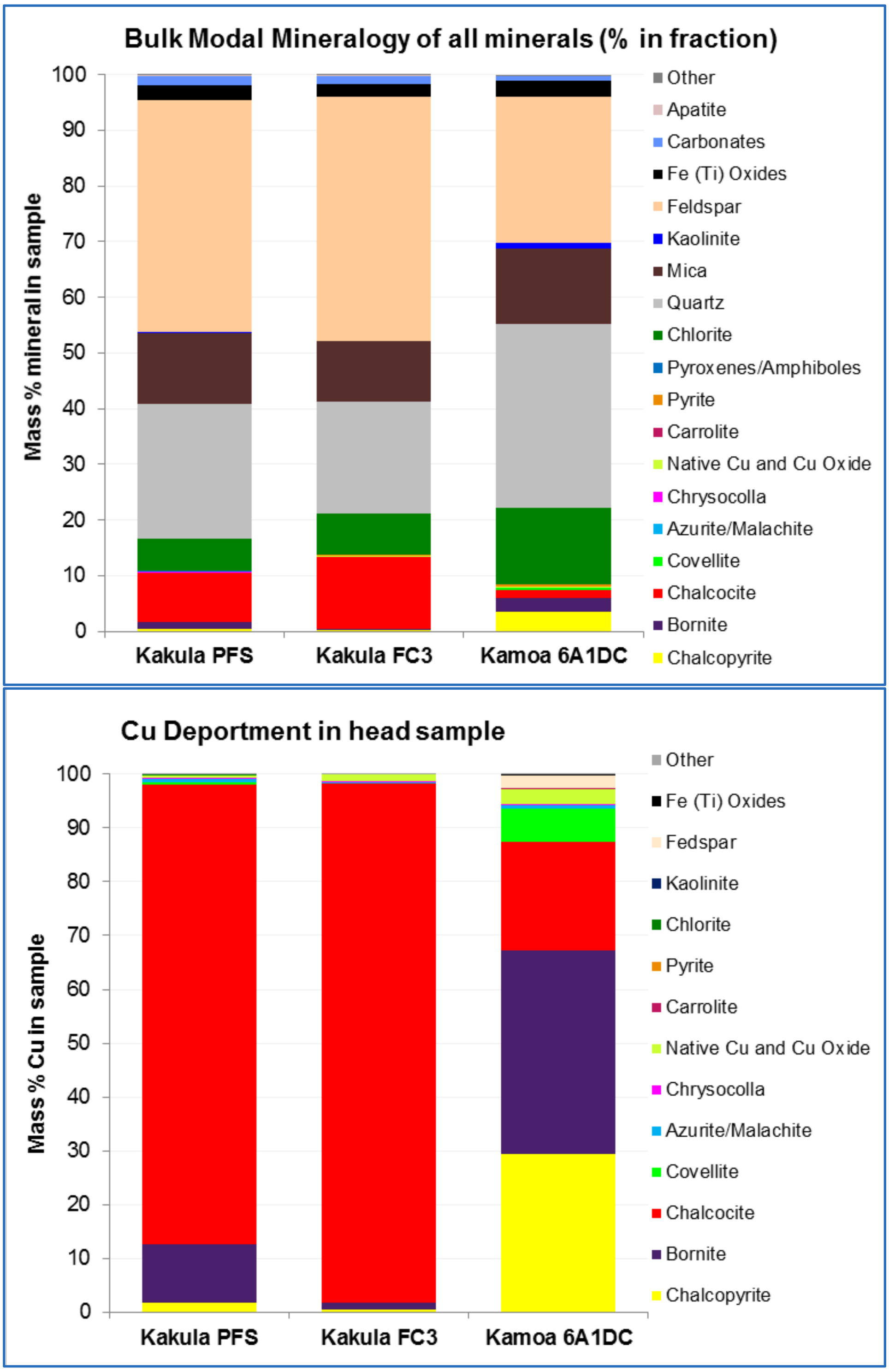


Figure 13.28 Cu Sulfide Grain Size Distribution Comparison Between Kamoa and Kakula

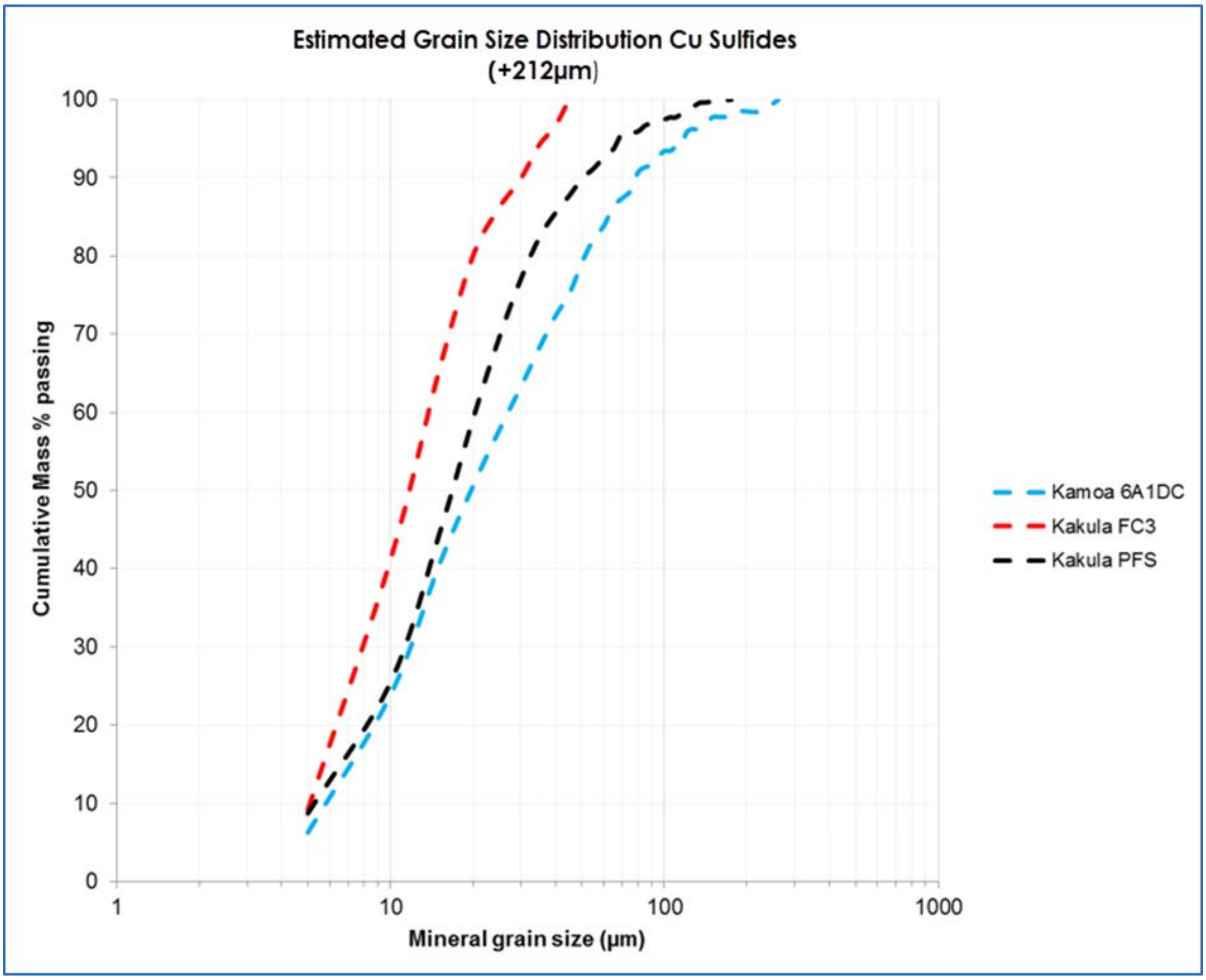


Figure provided by Ivanhoe, 2018.

13.3.2.1 Dominance of Chalcocite

The following was noted:

- The main Cu sulfide 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 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 FC1 sample sulfide 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 sulfide 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 sulfide 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 sulfide minerals as seen in Figure 13.29.

Figure 13.29 Comparison of Cu:S between Kamoa and Kakula Mineralisation

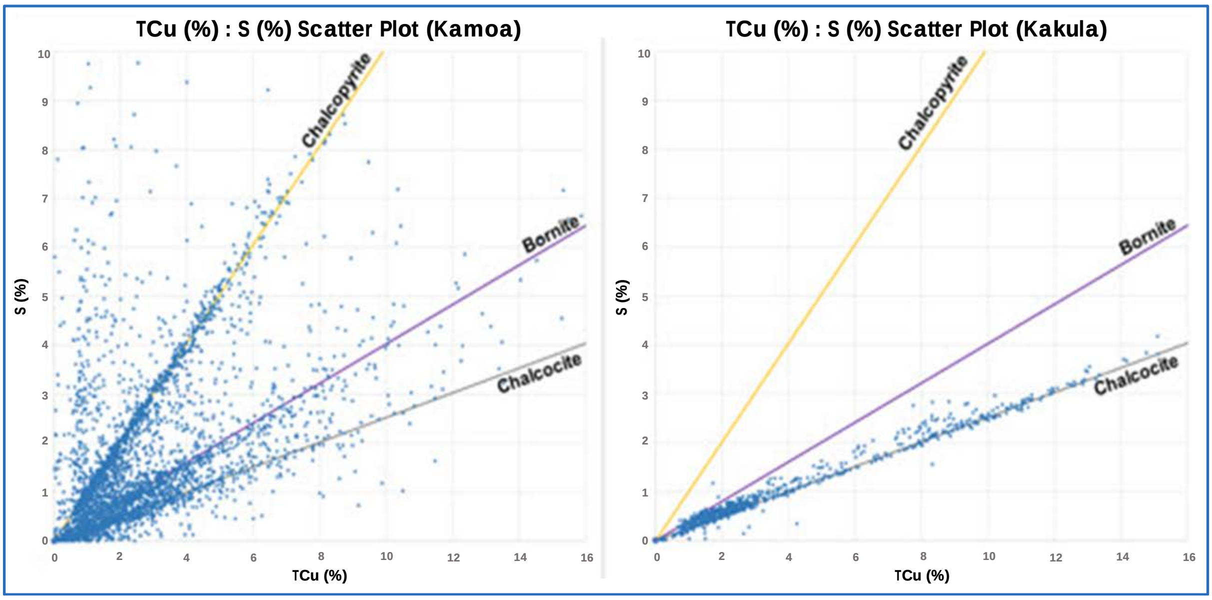


Figure by Ivanhoe, 2016.

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 sulfide 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 drill holes 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).
- 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	–	10.10	0.02	20.1	15.40
DD1017T SSL	125.4	–	9.80	0.01	22.7	17.10
DD1032T SDT	86.0	–	13.00	0.02	20.7	16.90
DD1032T SSL	77.0	–	10.90	0.01	19.0	15.40
DD1065T SDT	139.4	12.6	12.00	0.01	24.9	18.80
DD1065T SSL	214.3	–	10.70	0.01	24.5	19.10
DD997T SDT	197.9	–	11.80	0.05	20.0	17.60
DD997T SSL	107.7	12.7	11.20	0.03	24.1	19.50
FW SDT 1	81.0	–	13.50	0.06	20.7	17.90
FW SST 1	154.8	–	12.0	0.32	16.1	17.80
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 siltstone samples. The diamictite samples compared well to the earlier tested samples, however, the siltstone 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 siltstone 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 siltstone samples.

The BRWi results grouped the Kakula material in the hard–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 siltstone 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 22.6–46.5 (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 programmes.

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 flotation 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 Optimisation

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 aeration 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 regrind step from the scavenger cleaner feed to the scavenger recleaner feed.
- 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 re-cleaner 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 PFS Flow Sheet

Refer to Figure 13.30 for an illustration of the final Kakula 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 PFS Flow Sheet (Kamoa Copper SA, 2017)

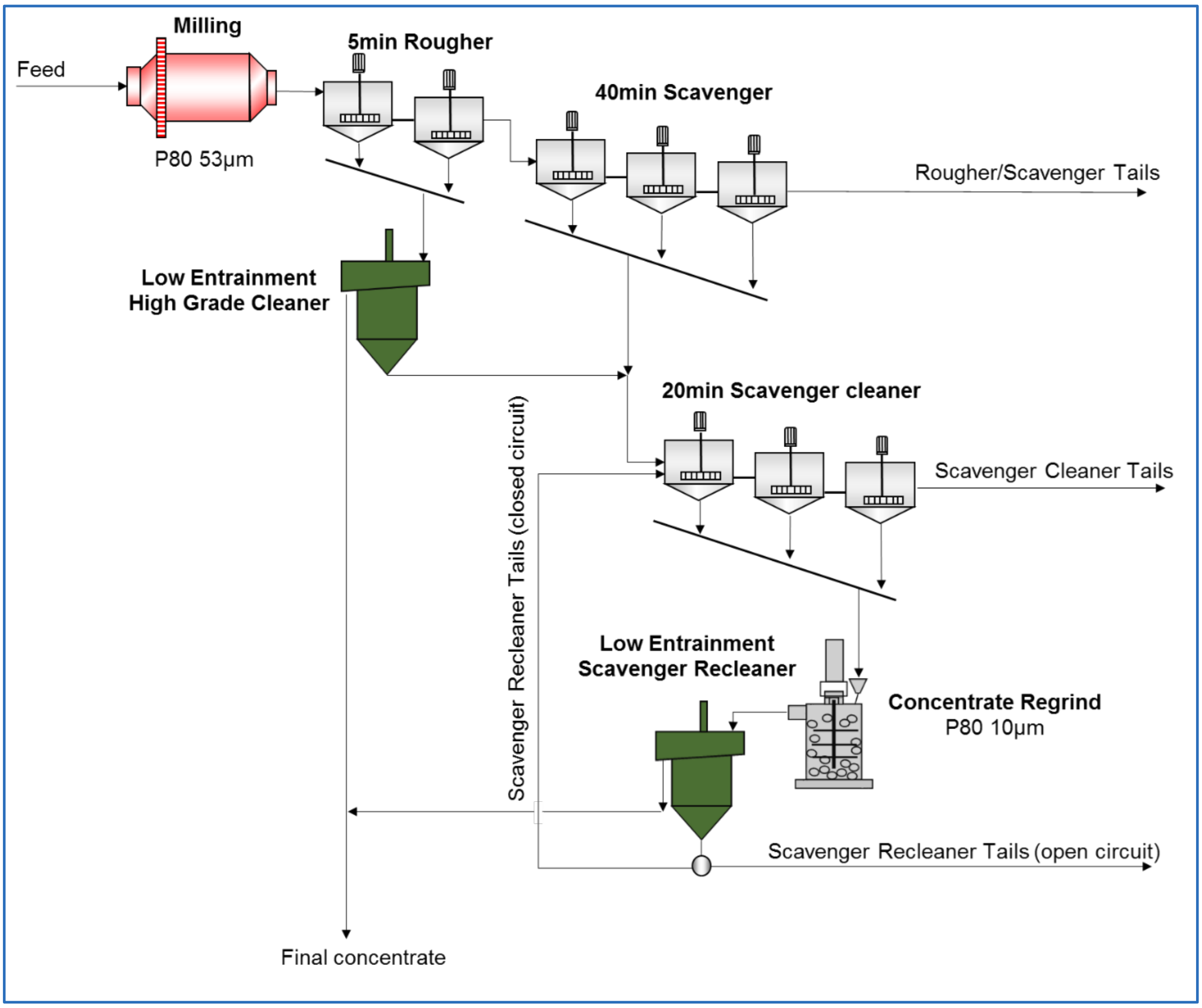


Table 13.24 Kakula PFS Flotation Parameters Summary

% Solids Grind	60
Grind Target	80% –53 µm
Grind Media	29.8 kg 440 SS (1" and ¾") rods
Grind Time	30:41

Stage	**Cell Size**	**Solids (%)**	**Est. (wt%)**	**SIBX**	**3477**	**SF22**	**Gas**	**RPM**	**Cum. Time**
Grind				179.0	32.0				
Ro Conc 1	4.5L	34%	~10–12%			76.0	5	1300	2
Ro Conc 2				26.0	5.0	9.0	5	1300	5
Ro Conc 3			~25%	17.0	3.0	16.0	9	1600	13
Ro Conc 4				17.0	3.0	16.0	9	1600	23
Ro Conc 5				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 Re-Clnr 1	2.5L		5%				3	1000	9
High Grade Re-Clnr 1	2.5L						2	1000	7
Scav Clnr 1	4.5L	12–13%	30%	0.0	0.0	10.0	5	1400	4
Scav Clnr 2				0.0	0.0		7	1600	13
Scav Clnr 3				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 Cleaner Conc 1–1	2.5L	8%	11%	12.4	2.2	10.0	3	1000	3
Scav Cleaner Conc 1–2						10.0	5	1000	8
Scav Cleaner Conc 1–3						10.0	6	1000	28
Scav Cleaner Conc 2	2.5L	3%	5%				6	1000	20
Scav Cleaner Conc 3	2.5L	3%	4%				5–7	1000	15
Total				310.2	55.6	203.0			

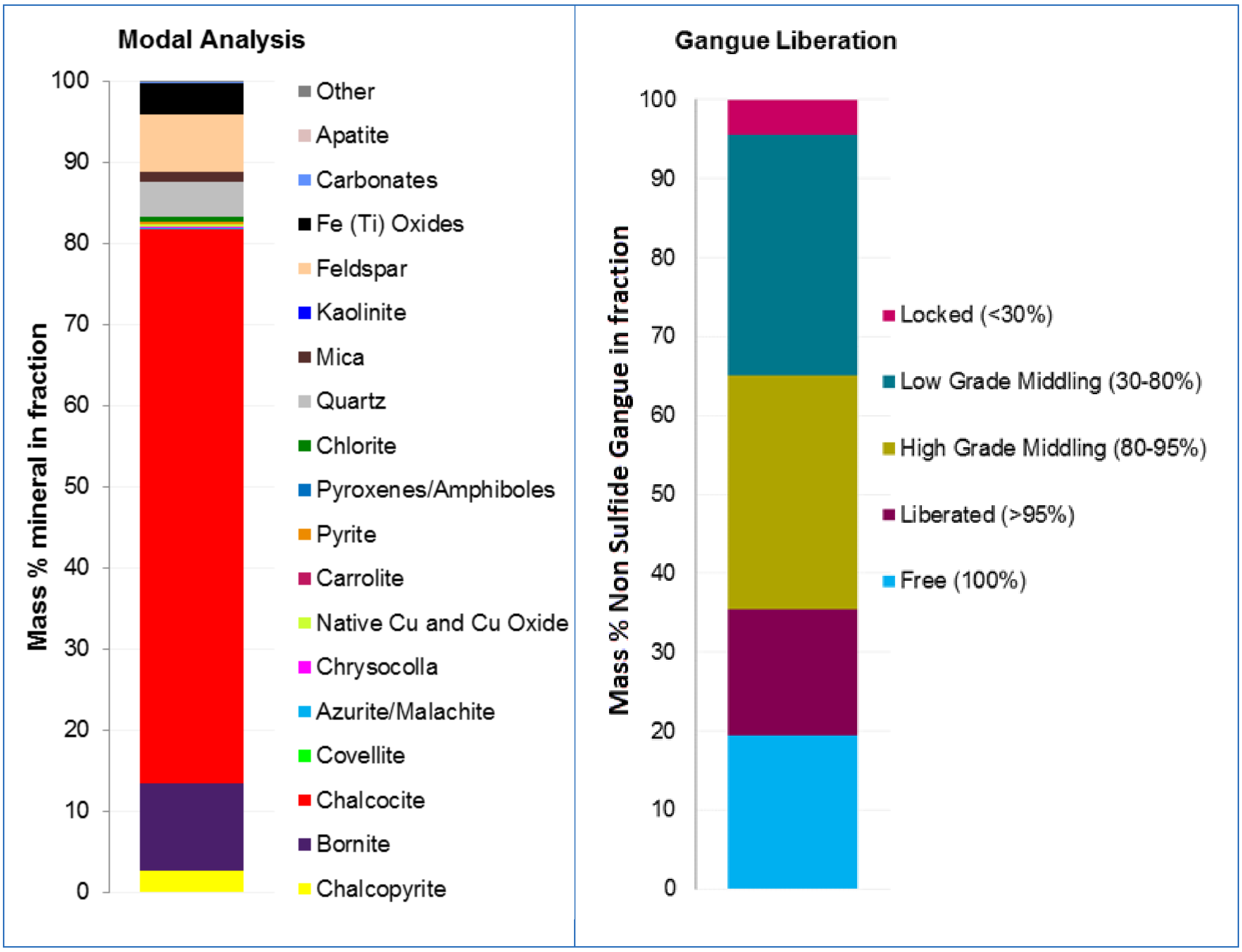
13.3.5.4 Flotation Products Mineralogy

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 sulfides, 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 3–5 µm. Almost 92% of the Cu sulfide minerals in the rougher tailings was locked – none of the Cu sulfide 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 sulfides of which 86% of the Cu as sulfides 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



OreWin

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	<0.80	<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	<0.70	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 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

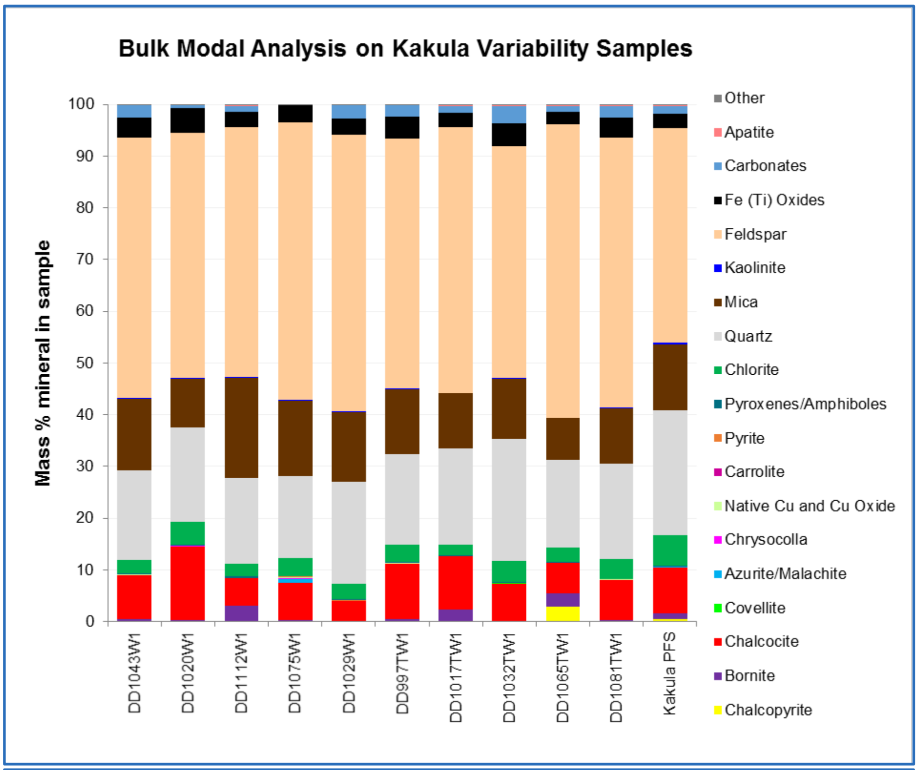
The following was noted from the head grade analysis:

- The samples tested varied from 2.6% Cu to 9.2% Cu, with sulfur 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 sulfide Cu at 86%.

Figure 13.32 Kakula Preliminary Flotation Variability Samples Mineralogy Summary



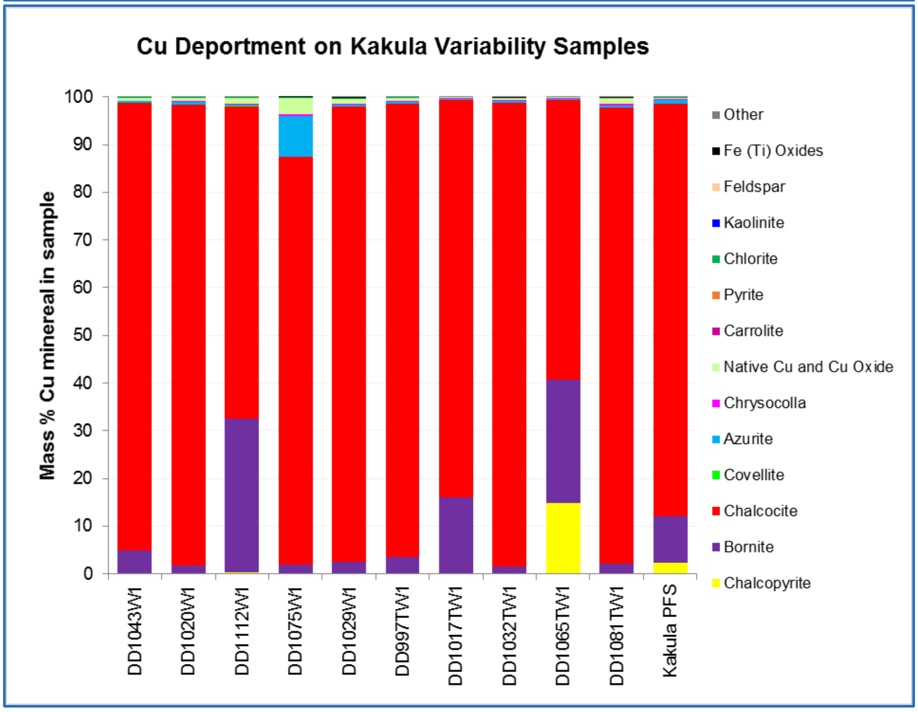
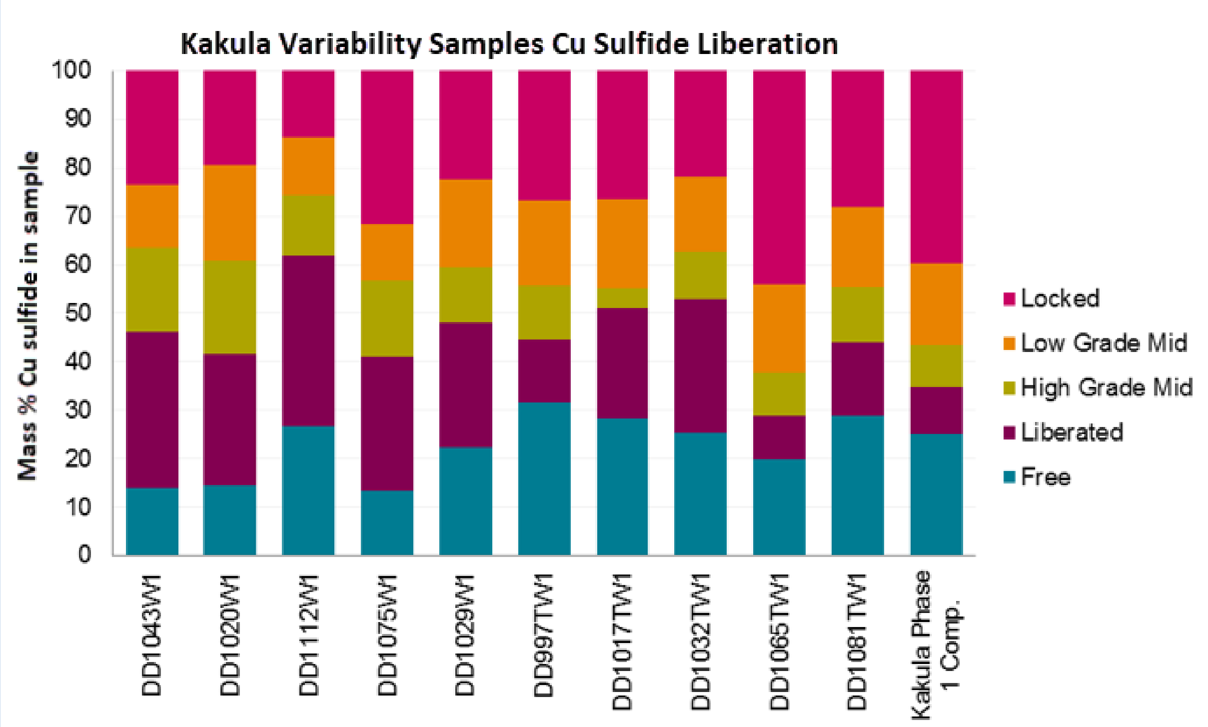


Figure 13.33 Kakula Preliminary Flotation Variability Samples Liberation Data



The Cu sulfide 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 sulfides. Liberation at a particle size of 80% –220 µm varied from 30–60% of the mass of Cu sulfides occurring as free or liberated grains, while the mass proportion of the locked Cu sulfides varied from 15–45%. The average Cu sulfide grain sizes varied significantly from 8–20 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 PFS flow sheet. Collector dosages were adjusted to a maximum of 50g/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-sulfide 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-sulfide Cu are not consistent across the Kakula ore body. 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.
- 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, 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/m²$\cdot$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/m².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, 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 P80	48µm	41µm
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	33mg/L
0.22 $m^2/(t/d)$	45	25	50mg/L
0.22 $m^2/(t/d)$	40	20	99mg/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 (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 14–31 mm, while the resulting solids throughput ranged from 578–977 $kgDS/m^2.h$. The residual cake moisture varied between 15.9–18.6% solids (w/w).

13.3.8 Additional Testwork on Kakula Resource

13.3.8.1 Mini-Pilot Plant Campaign

During March 2019, XPS conducted a mini-pilot plant campaign, to generate sample for the following testwork campaigns:

- 400 kg of final tailings material for backfill testing.
- 30 kg of scavenger cleaner concentrate for regrind testwork.
- 20 kg of high-grade rougher concentrate for Jameson cell testwork including Jameson Cell testwork, conducted by XPS.

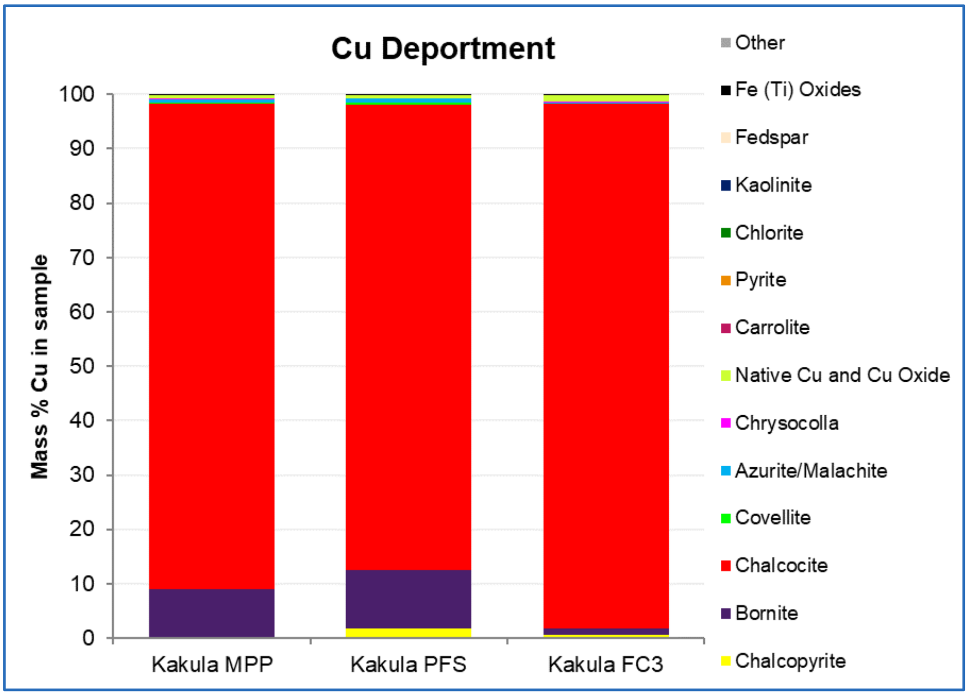
The above mini-pilot plant testwork further demonstrated the performance of the flow sheet developed during the prefeasibility study. Open circuit batch tests, locked cycle testing, and mineralogical analyses were completed to match the metallurgical performance of the mini-pilot plant sample to the prefeasibility results, prior to the pilot plant campaign.

The average head grade of the MPP sample of 7.09% Cu was roughly 1% higher than the PFS development composite sample.

Mineralogical Assessment

Feed mineralogy was conducted at P_{80} 212 µm for comparison against the PFS Kakula samples (Figure 13.34).

Figure 13.34 MPP Sample Mineralogy Compared to Previous Kakula Samples

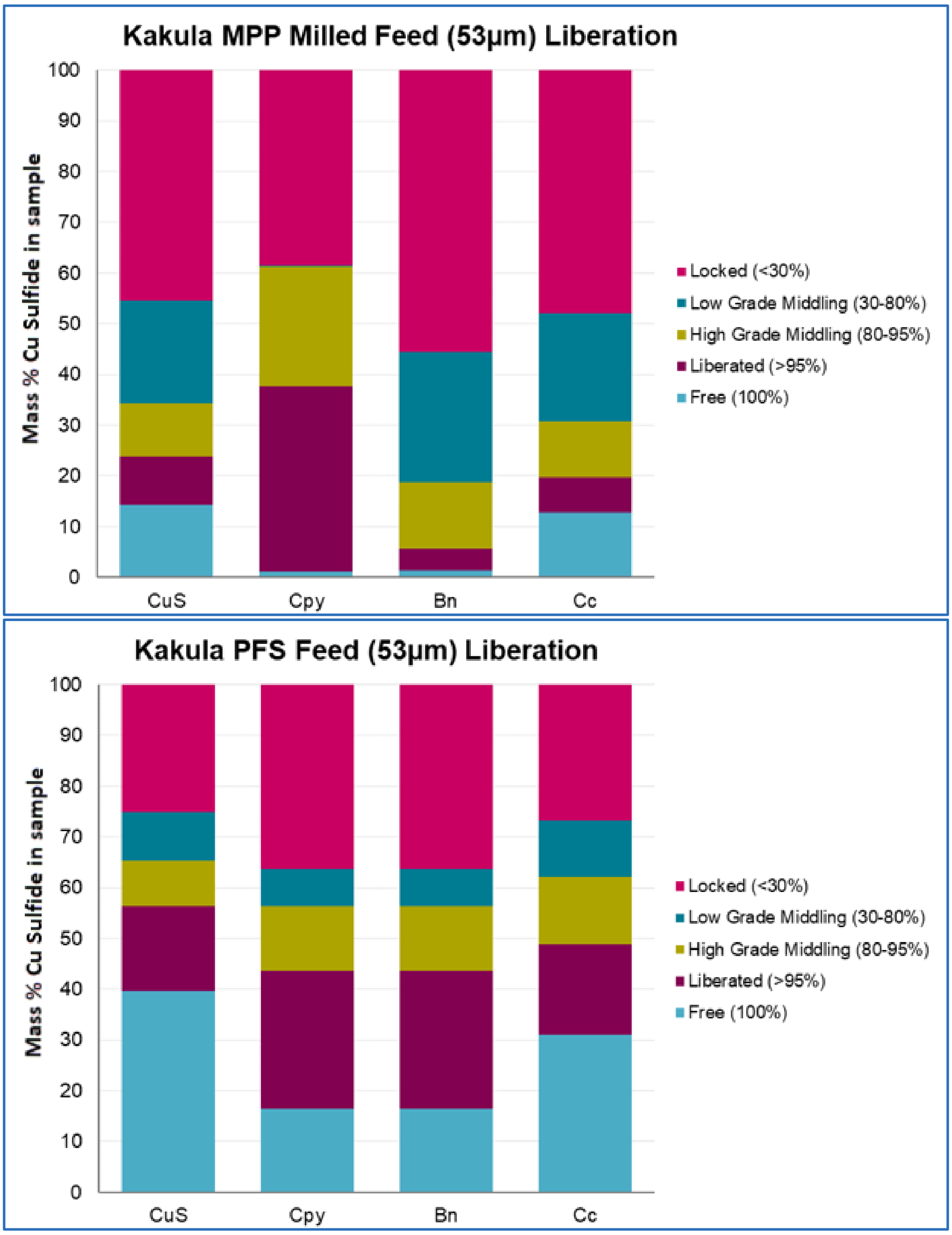


The following was noted:

- Overall, the MPP sample's mineralogy was similar to the PFS development composite sample
- Modal mineralogy indicated that the MPP sample contained 12% Cu sulfide which consisted mainly of chalcocite (89%), and bornite (8.8%).
- The MPP sample revealed very low levels of chalcopyrite (0.25%).
- Liberation was poor above 53 µm for all the Cu sulfides.

A milled feed sample at 53 µm was analysed (unsized) to evaluate the liberation at the target rougher feed grind. The results are summarised in Figure 13.35 below and compared against the PFS sample.

Figure 13.35 Cu Sulfide Liberation at 53 µm Grind



The above indicates that the liberation of the MPP sample was not as well liberated as the Kakula PFS sample, and that the total locked Cu sulfides was measured at 45% compared to a PFS sample of 25%.

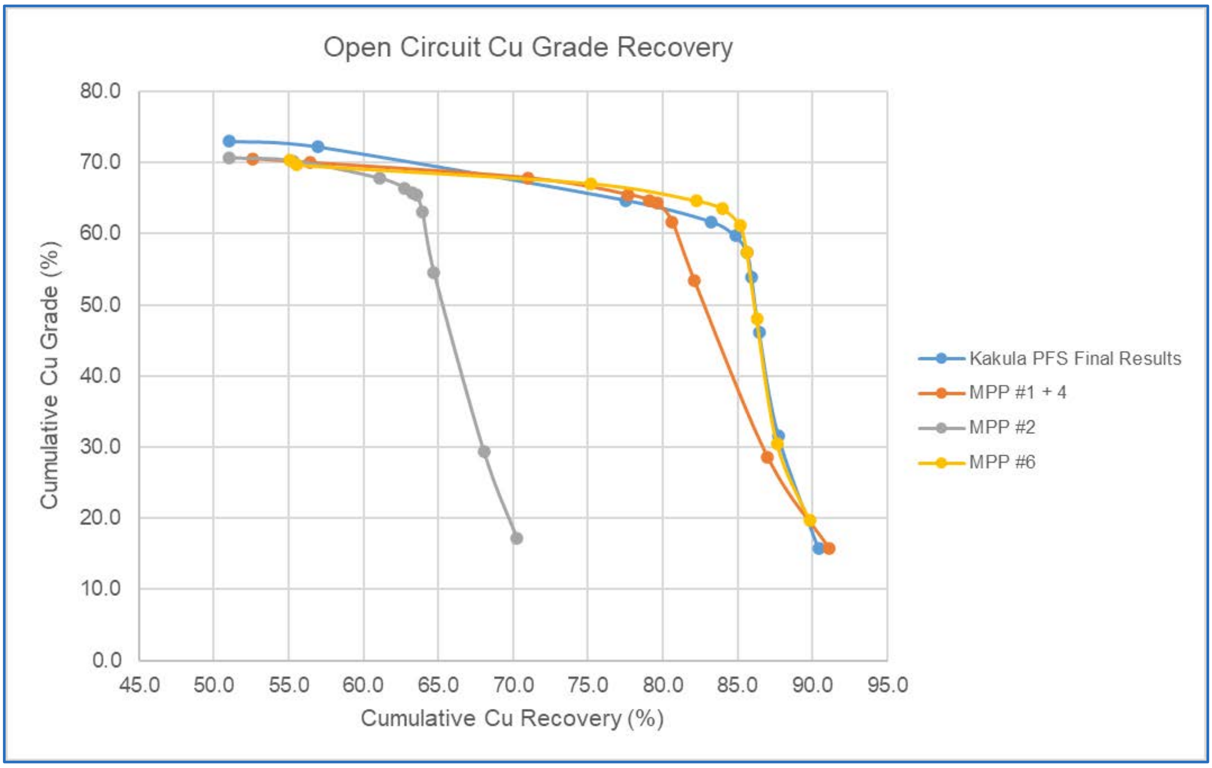
Open Circuit Cleaner Testwork

Duplicate open circuit cleaner tests (MPP #1, and MPP #4) were performed to baseline the MPP composite against the PFS flow sheet. The duplicate test reported a rougher grade and recovery in line with the PFS results, however, the scavenger cleaner circuit reported higher Cu losses. The final concentrate Cu recovery was noted as 79.6% at 64.3% Cu and 8.9% SiO_2.

Another test, MPP #2, was conducted without the high-grade cleaner tailings moving forward to the scavenger cleaner circuit, to determine if the scavenger cleaning circuit could perform without the high-grade tailings contribution. The results indicated a similar shaped grade-recovery curve to the duplicate tests, with a recovery offset of roughly 20% (Figure 13.36). The offset is due to the high-grade cleaner tailings not reporting to the scavenger cleaner circuit. The scavenger concentrate, at a much lower grade, was able to upgrade to final concentrate grade which indicates that there would be no risk to the planned locked cycle test by feeding the high-grade cleaner tailings to the next cycle.

A fourth open circuit cleaner test, MPP #6, was conducted during which 25% higher collector dosages was applied to increase Cu recovery. The increase in collector dosage was motivated by the higher sample feed grade. The increased in collector resulted in a final recovery of 85.6% Cu at a final product grade of 57.3% and 14.9% SiO_2 (Figure 13.36).

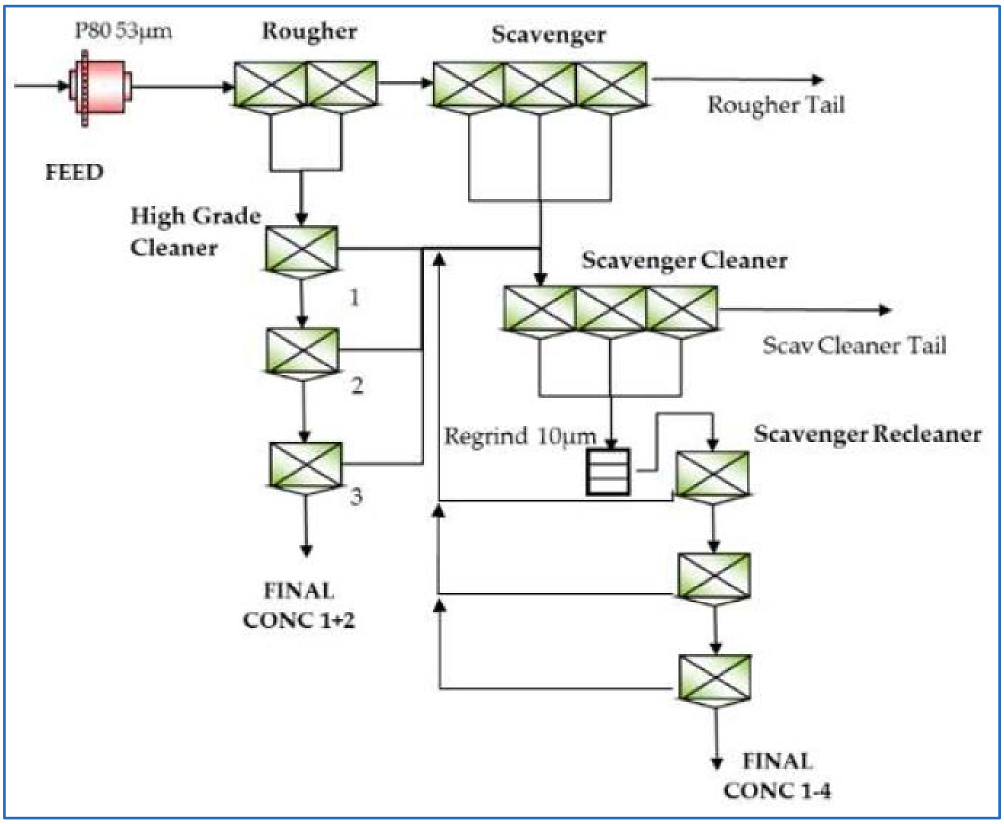
Figure 13.36 MPP Cleaner Circuit Testing Compared to PFS Results



Locked Cycle Testing

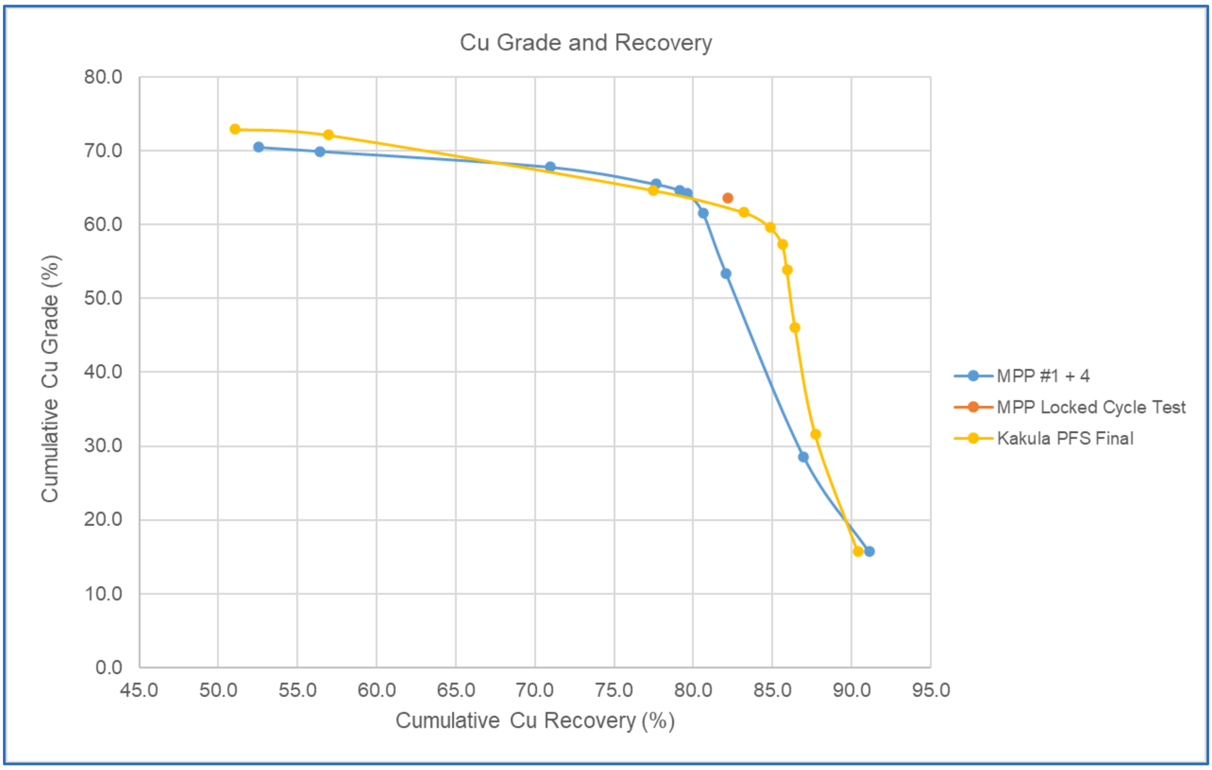
A single, 6-cycle locked cycle test was conducted to determine the effect of recirculating the scavenger recleaner tailings back to the scavenger cleaner (Figure 13.37).

Figure 13.37 MPP Locked Cycle Test Flow Sheet



A total recovery of 82.2% Cu, at a final product grade of 63.6% Cu and 9.9% SiO_2 was recorded. Copper lost to the rougher / scavenger tailings was noted as 8%, and in line with the open circuit tests on the same sample. The Cu losses to the scavenger cleaner tailings was slightly lower compared to the open circuit test (9.8% compared to 11.5%). Overall, the locked cycle test increased the Cu recovery by 2.6%, compared to the MPP open circuit runs #1 and #4, at an increase of 1% SiO_2 grade in the final product. Refer to Figure 13.38.

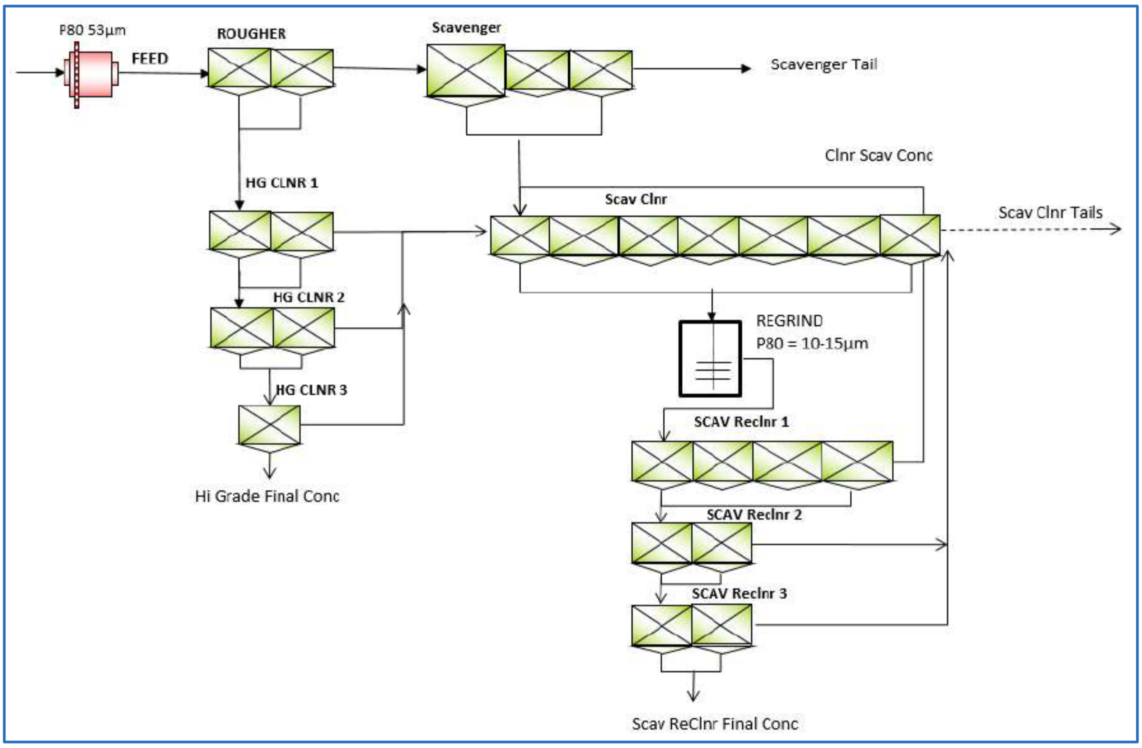
Figure 13.38 MPP Locked Cycle Result Compared to Open Circuit Testing



Backfill Tailings Sample Generation

The first MPP run was aimed at producing a combined tailings product for backfill testwork utilising the flow sheet in Figure 13.39.

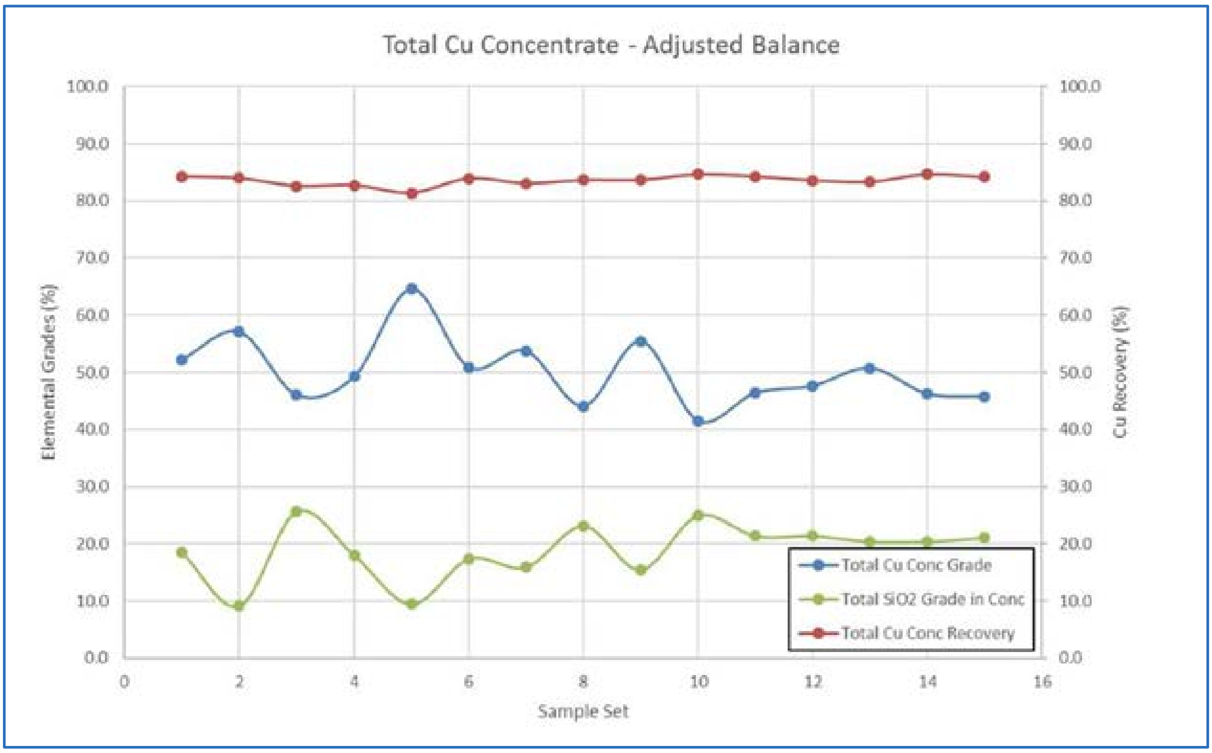
Figure 13.39 MPP Run #1 Flow sheet to Produce Combined Tailings Sample



Source XPS (2019)

MPP run #1 ran for roughly 58 hrs, feeding just under 600 kg of fresh feed. Final products were sampled every 4 hrs and assayed to produce a mass and metal balance. The data from MPP run #1 is summarised in Figure 13.40.

Figure 13.40 Mini-Pilot Plant Run #1 Performance



Source XPS (2019)

Final product Cu grades varied significantly during the first half of the run in attempt to achieve final grade and to improve overall recovery. The Cu recovery to final product was stable around 84%. Recovery was limited by losses to the scavenger tailings and can be improved with additional scavenger flotation capacity. It is noted that the MPP run scavenger cleaner circuit residence times were lower than targeted.

Scavenger Cleaner Concentrate Sample Generation

Following the first MPP run, the regrind and scavenger recleaner circuits were taken offline to start collecting the scavenger cleaner concentrate as a product. MPP run #2 run took roughly 18 hrs to complete, during which a total of 25 kg of scavenger cleaner concentrate was collected. This sample was filtered, dried, and shipped for regrind testing. A second, smaller, sample was collected over 4hrs to provide feed to a single Jameson Cell test.

Open circuit sampling results indicated that the scavenger cleaner concentrate mass flow varied between 13–19%, at a concentrate grade between 10–18% Cu.

High Grade Cleaner Jameson Cell Testwork

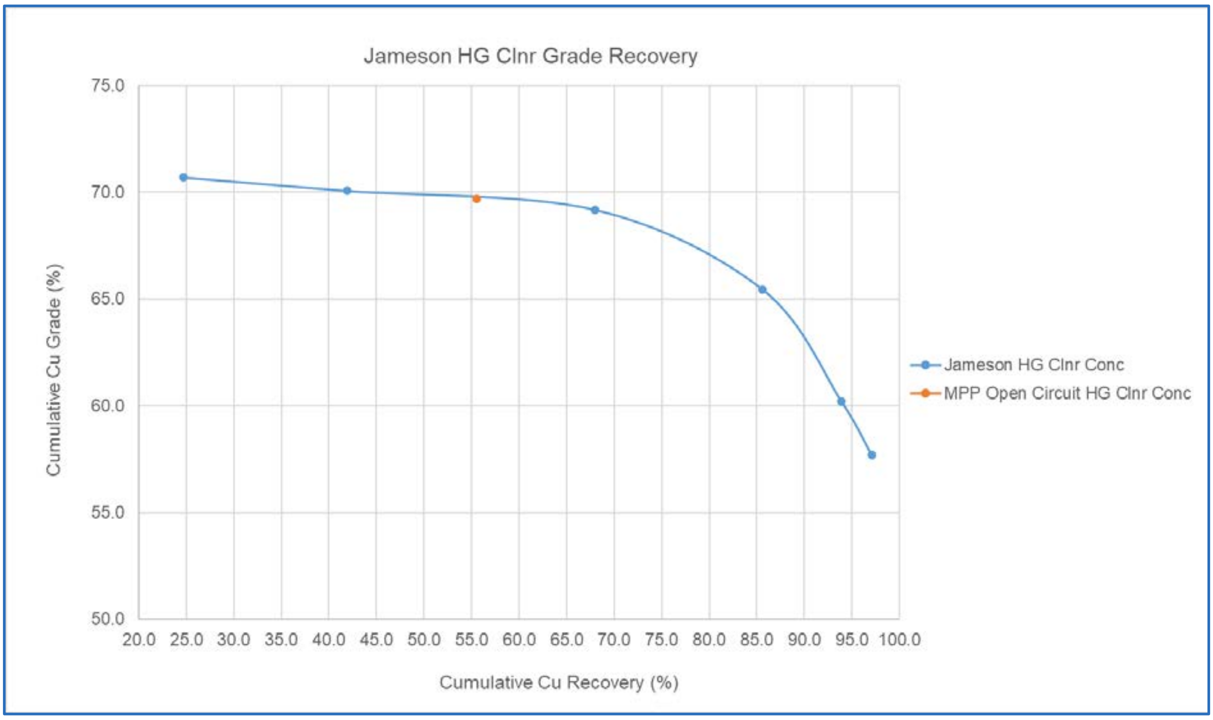
A final MPP run was conducted to produce high grade rougher concentrate sample for Jameson cell testing, utilising a truncated flow sheet of the mainstream flotation circuit only.

Rougher concentrate product produced during the third MPP run was used to demonstrate the scale up of the low entrainment cleaning during bench scale testing, to the performance using a pilot Jameson Cell unit.

The High-Grade Jameson cell upgraded the feed from 40.8% Cu to 57.7% Cu, recovering over 97% of the Cu (Figure 13.41).

The Jameson cell test compared well against the benchmark set in the open circuit tests. Cu concentrate grade maintained above 69% over the first three concentrate increments. This single test indicated that the Jameson cell performance will be able to match the results produced in the bench scale dilute cleaning tests.

Figure 13.41 Jameson High Grade Cleaner Cu Grade-Recovery Curve

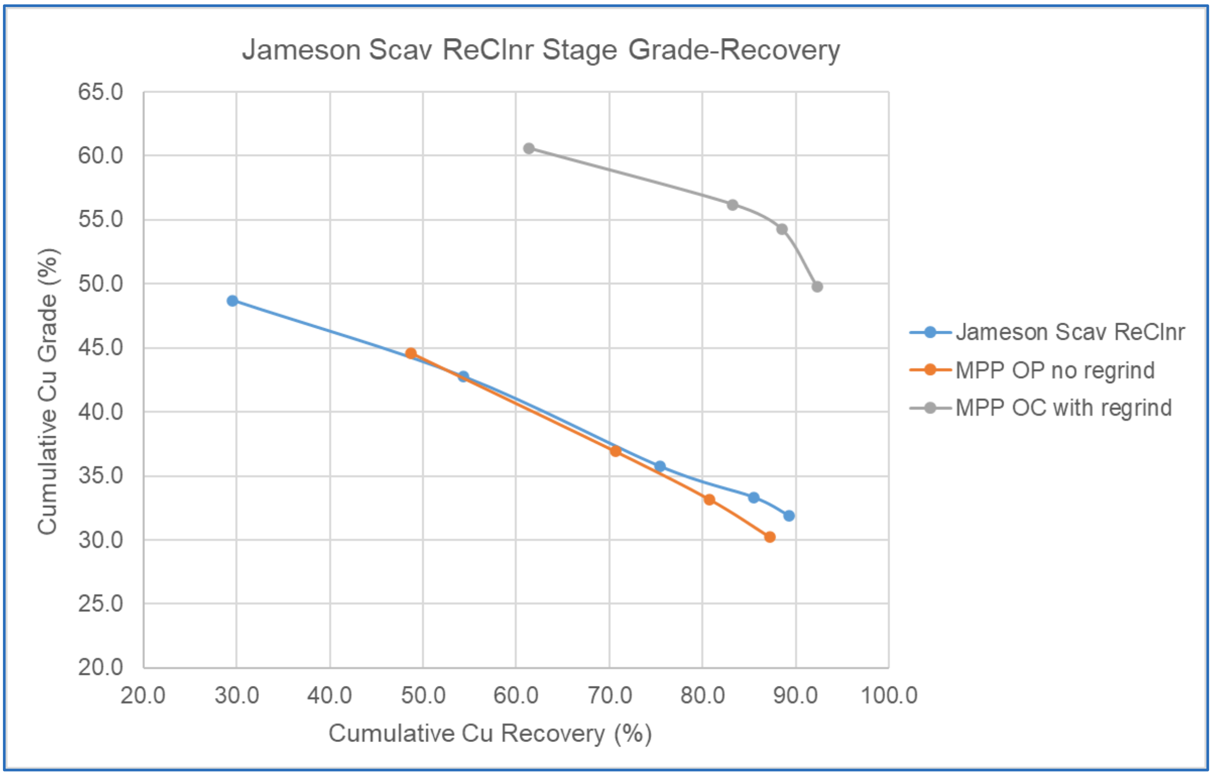


Source XPS, 2019

Scavenger Recleaner Jameson Cell

A third Jameson cell test was conducted, utilising the sub sample produced during the Mini-pilot plant run #2. The intent of this test was to test dilute cleaning without a regrind step (Figure 13.42).

Figure 13.42 Jameson Scavenger Recleaner Cu Grade-Recovery Curve – With and Without Regrind



Source XPS, 2019

The scavenger recleaner Jameson cell upgraded the feed from 18.1% Cu to 31.9% Cu, recovering just under 90% of the Cu. The Jameson cell run without regrind matched the open circuit test which excluded the regrind step. The exclusion of the regrind step resulted in a much lower product grade and recovery.

It is not recommended to process the Kakula material without the regrind step.

13.3.8.2 Tailings Settling Testwork

Outotec was commissioned to conduct thickening testwork on the flotation tailings to determine the thickening properties and to confirm final tailings thickener design as a process guarantee.

Sample characterisation indicated a P_{80} of 50 µm, and a solids specific gravity of 2.86. Testing recommended a design flux of 0.42 t/h/m^2 to produce an underflow product of 57% solids (w/w) when dosing 30 g/t SNF 45 VHM and 60 g/t SNF 910 SH, with an overflow clarity of <100 mg/l.

13.3.8.3 Concentrate Regrind Testwork

Grinding Solutions Ltd (GSL) was contracted by Metso in March 2020 to conduct hydro-cyclone and signature plot testwork on a Kakula scavenger cleaner concentrate sample, as prepared during the mini-pilot plant campaign, in support of a contractual process guarantee to be offered by Metso for the supply of the concentrate regrind mills to the Kakula Phase 1 project.

Multiple test runs were conducted using 1-inch and 2-inch cyclones, to achieve the targeted overflow particle size distribution. The 1-inch units achieved a cut size of P_{80} 5.8 µm while the 2-inch unit produced an overflow P_{80} 8.4 µm. The results from the bulk cut conducted using a 2-inch unit is summarised in Table 13.35.

Table 13.35 GSL 2-Inch Hydro Cyclone Performance Summary

	Density (kg/L)	Solid Content (% w/w)	Mass Split	P_{50} (µm)	P_{80} (µm)
Feed	1.20	25	100	5.4	30.1
Overflow	1.07	9.2	73	3.7	8.4
Underflow	1.86	68.5	27	46.3	80.9

The signature plot was carried out on the GSL laboratory stirred media detritor using 3 mm Kings 3 SG grinding media. The cyclone underflow was diluted from 68.5% solids (w/w) to 50% solids (w/w). The power model indicated that 23.79 kWh/t was needed to achieve a grind size of P_{80} 10 µm, however, considering the cyclone overflow cut size and mass split the regrind step would only require 20.14 kWh/t to achieve a combined product of P_{80} 10 µm. The signature plot summary is shown in Table 13.36.

Table 13.36 Signature Plot Summary

kWh/t	0	3.9	7.4	11.0	14.7	22.1	29.4	44.2
P_{80} µm	76.6	38.3	22.4	15.2	12.3	10.1	8.7	7.5

13.3.8.4 Flotation Tests using Underground Mine Water

XPS was contracted by Kamoa Copper SA in October 2019 to perform several batch flotation tests to examine if excess underground mine water could be used as process water make-up without prior treatment.

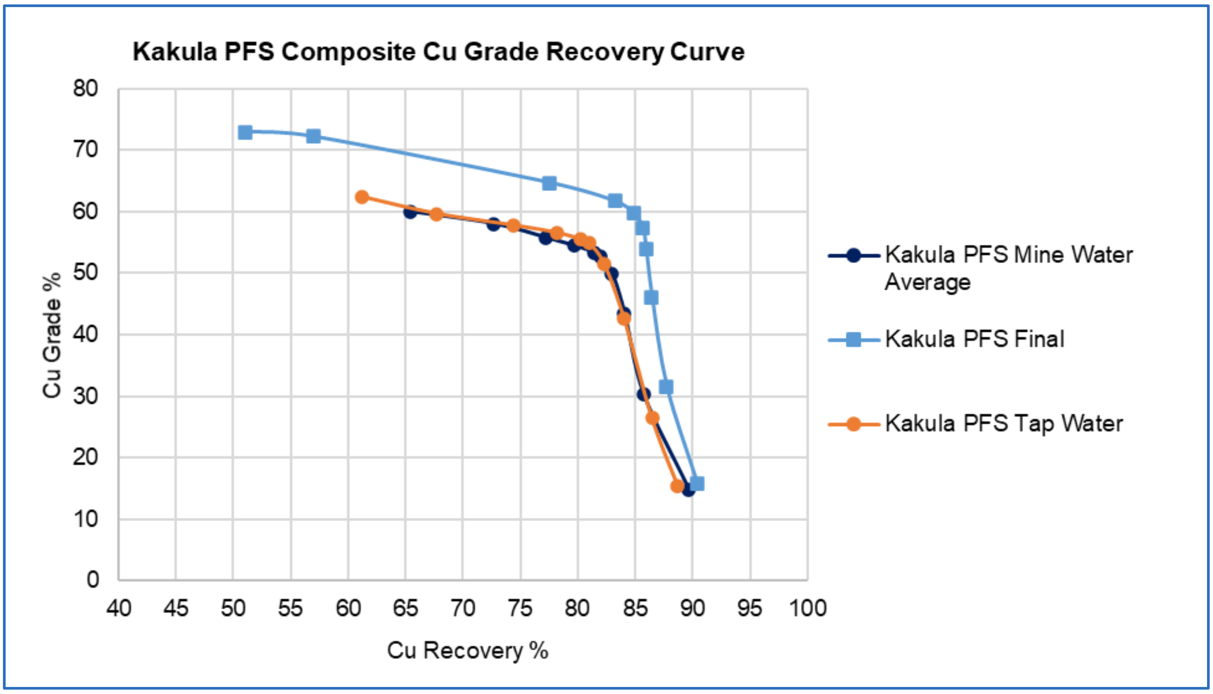
Table 13.37 Flotation Results using Mine Water

Sample	Water	Cu Recovery (%)	Cu Concentrate grade (%)	SiO_2 Concentrate Grade
Kamoa P6A Signature Plot Composite	Tap	86.6	36.2	13.0
	Mine	86.6	34.5	14.4
Kamoa P6A Supergene Composite	Tap	73	41	30.0
	Mine	73	41	30.0
Kakula PFS Composite 2019 testing	Tap	81.0	54.8	14.2
	Mine	82.0	52.8	16.7

A baseline test on the Kakula PFS composite sample was conducted using XPS tap water, which was used to compare the outcome of the mine water test against.

Figure 13.43 indicates similar Cu recovery and grades independent of the water type used.

Figure 13.43 Effect of Mine Water Flotation Testing on Kakula PFS Composite Sample



Source XPS, 2019

The recoveries achieved during this testing was lower compared to the PFS testwork campaign, which was attributed to aging and oxidation of the high chalcocite sample.

13.3.9 Kakula Recovery Estimate

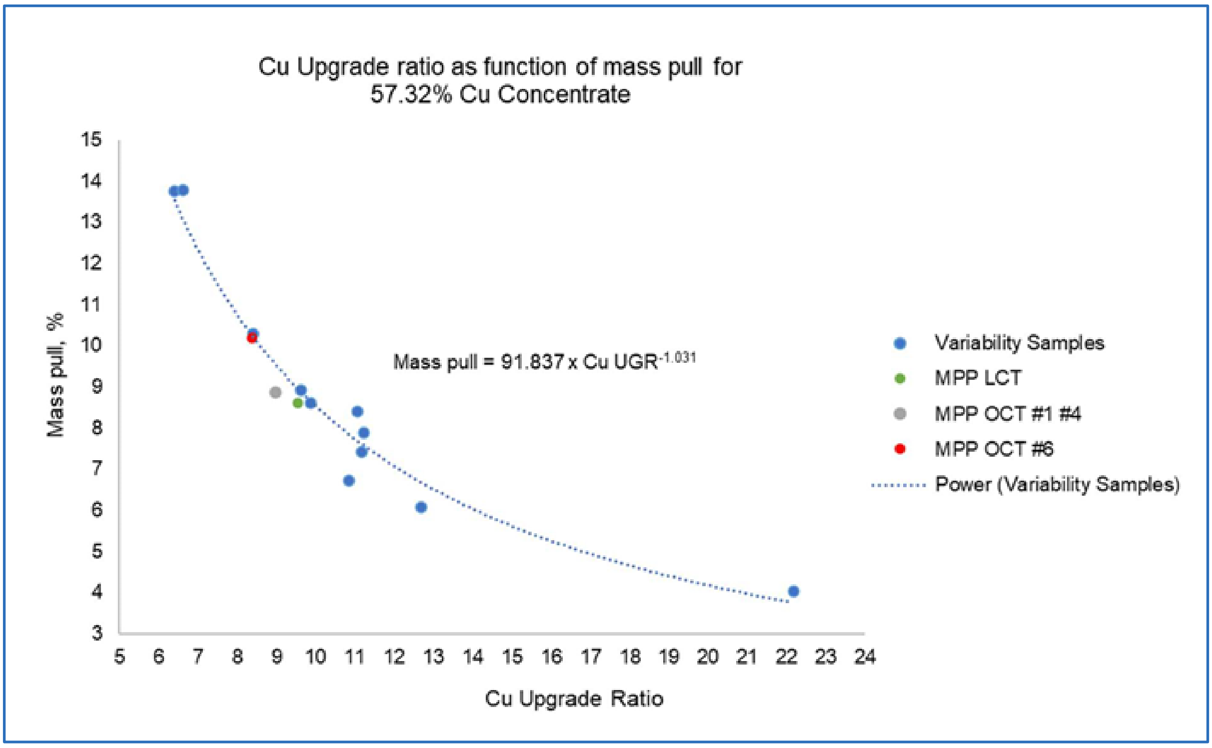
The recovery estimate for the Kakula feasibility study is based on the test information generated by the Kakula PFS campaign and the Kakula variability campaign.

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 13.44.

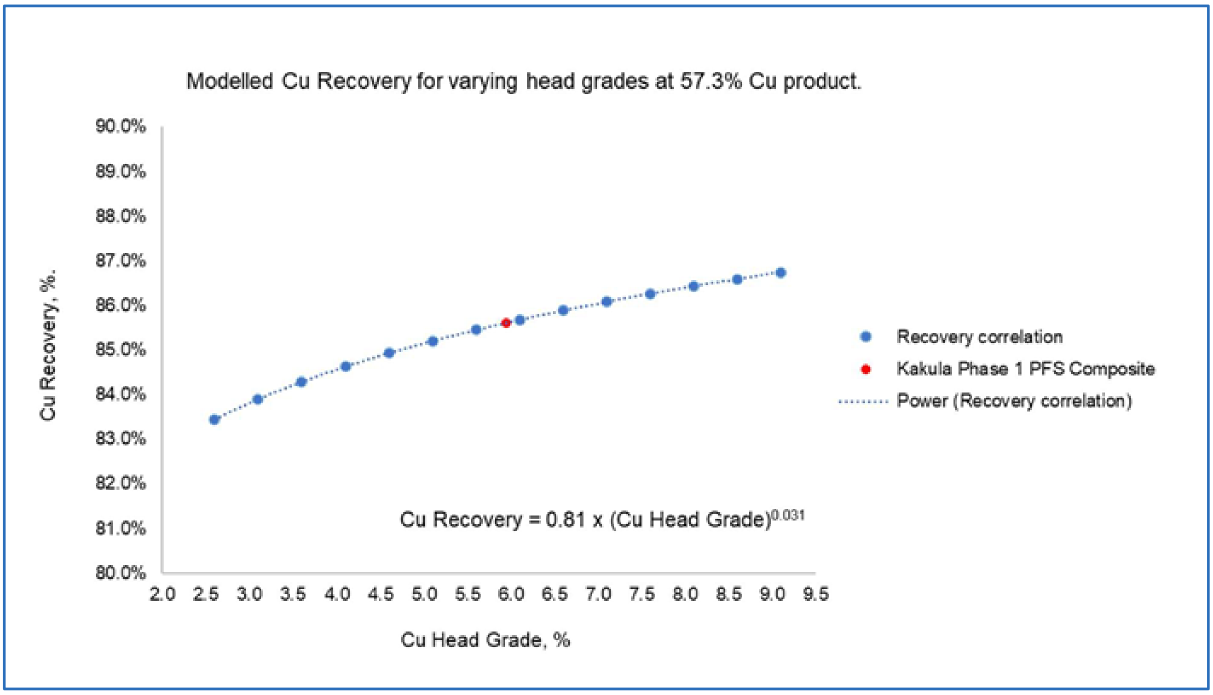
The datapoints from the locked cycle test and open circuit tests conducted during the mini-pilot plant campaign correlates well with the variability testwork data.

Figure 13.44 Kakula Copper Upgrade Ratio verses Mass Pull



The resulting correlation from Figure 13.44 was used to 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. The Kakula Cu recovery algorithm is shown in Figure 13.45.

Figure 13.45 Kakula Cu Recovery as a Function of Cu Head Grade

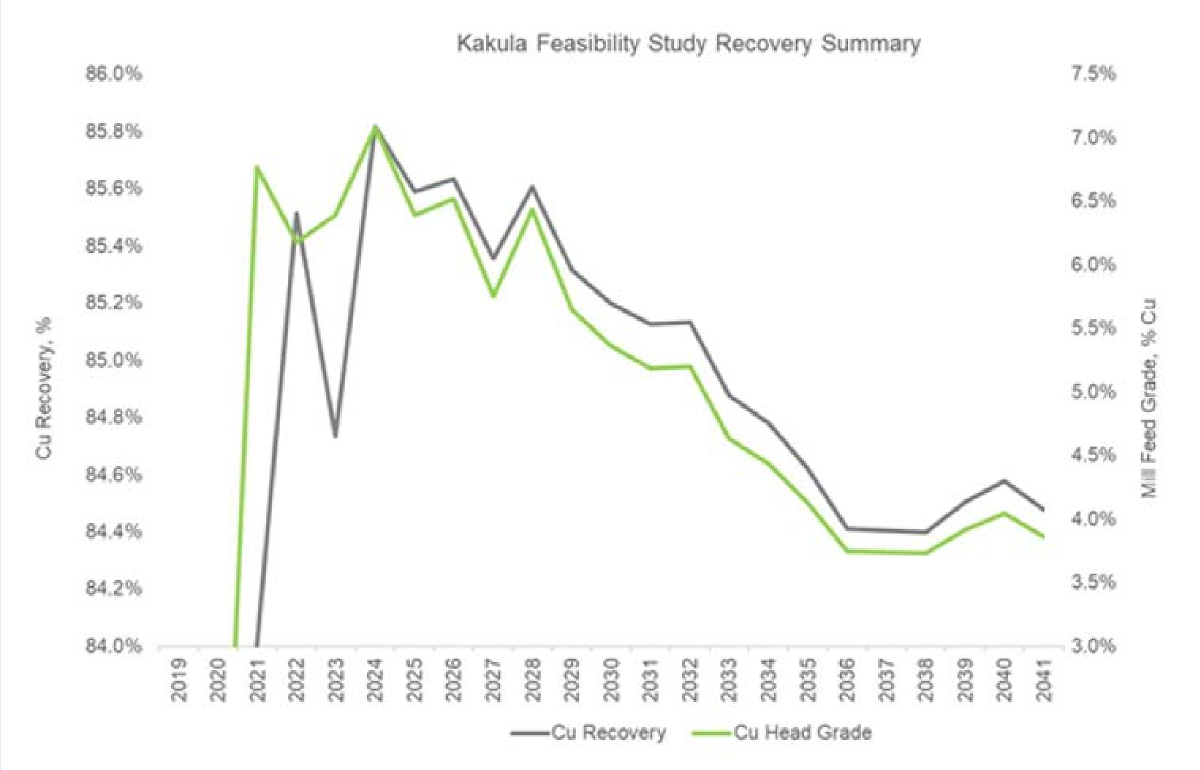


The mass pull and recovery correlations were applied to the Kakula feasibility study life of mine production plan. The following discounts were applied to allow for instabilities during ramp-up:

- 2021 recoveries were discounted by 2% to cater for ramp-up of the first production module.
- 2023 recoveries were discounted by 1% due to the ramp-up of the second production module.

The average Cu recovery over life of mine to produce a 57.32% Cu concentrate was calculated at 85.23% from a 5.22% Cu head grade. The life of mine recovery summary is presented in Figure 13.46.

Figure 13.46 Kakula Feasibility Study Recovery over Life-of-Mine



13.4 Testwork on Kakula West

13.4.1 Preliminary Testwork on Kakula West Material

In 2018, XPS conducted mineralogy and flotation tests on a single Kakula West composite sample.

In 2018, XPS conducted mineralogy and flotation tests on a Kakula West composite sample grading 3.17% Cu. 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.

13.4.2 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 13.38. 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 13.39.

Table 13.38 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 13.39 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

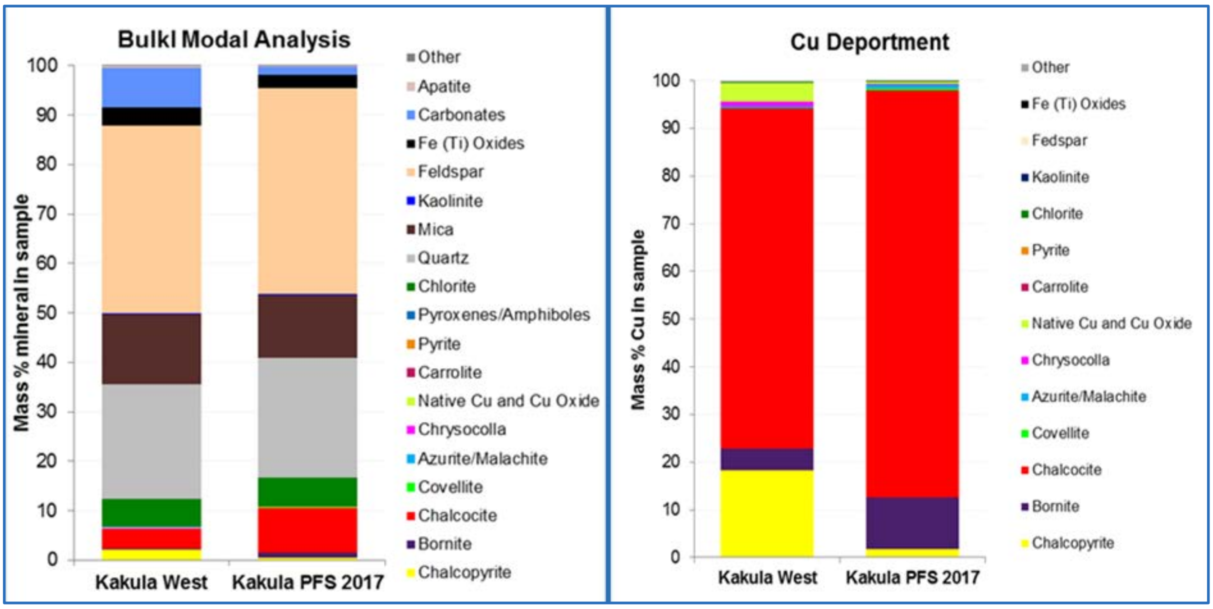
A summary of the bulk modal analysis and Cu deportment study conducted on the Kakula West sample, at 80% passing 212 µm, is given in Figure 13.47.

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 sulfide minerals was noted as similar to the Kamoa Phase 6 sample – slightly coarser than the Kakula PFS sample tested.

OreWin

Figure 13.47 Kakula West Sample Mineralogy



13.4.3 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 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 5.6–4.8%).
- 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 5.0–8.6%).

The testwork, however indicated 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 prefeasibility and feasibility level process design respectively. 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 sulfides in all Kamoa samples leads to strong recovery of silica through attachment with these sulfides. 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 14–15% 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 levels 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 Kakula regrind power requirement has been confirmed by testwork as described in Section 13.3.8.3.

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.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The Kamoa and Kakula Mineral Resource models are two separate models within the Project area.

The resource estimation methodology combines stratigraphic and mineralised units to construct a full three-dimensional (3D) block model with multiple horizontal domains stacked vertically. Mineralised zones are defined using an approximate cut-off grade of 1% TCu (locally 0.5% TCu), and a minimum 3 m vertical thickness was required for reporting the Mineral Resource to reflect the minimum underground mining height.

At Kamoa, six mineralised domains were modelled within different stratigraphic horizons, including the Bonanza Zone mineralisation which is hosted within the KPS. At Kakula, a single mineralised zone was modelled near, or just above, the Roan (R4.2) contact, which is locally separated into two domains based on whether the host is the basal siltstone or the overlying diamictite unit.

To account for the undulations of the deposits and ensure that the vertical 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). To prevent smoothing of grades vertically during estimation, selection of samples used for both the variography and grade estimation were constrained vertically from 25% to 30% of the vertical dilated thickness to preserve the vertical grade profile and mineralogical zonation. 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.

Collar, survey, assay, stratigraphy and SG data were exported from the Ivanhoe acQuire database as a series of csv files, imported into Datamine RM mining software, and combined to form a desurveyed drillhole file for each deposit area.

14.2 Selective Mineralised Zones

14.2.1 Kamoa

The Mineral Resource estimate used 998 drillhole intercepts, which include drillholes within the mining lease, but excludes drillholes within the Kakula, 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 998 drillholes are 17 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.

In general, the selective mineralised zone (SMZ) is based on a 1% TCu cut-off. The basal contact of the SMZ is usually sharp and easily defined. In areas with gradational vertical 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 key considerations 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 drillholes.

The different SMZs occupy distinct positions vertically, and lateral extents are largely controlled by the basin structures especially at Kansoko Sud and along the Bonanza Zone fault. The most laterally extensive SMZs are those hosted within the basal diamictite. The Upper SMZ is developed north-west of the Kansoko Sud growth faults, and is the most laterally continuous and best developed of the modelled minzones, hosting the majority of the estimated Kamoa Mineral Resources. The Upper SMZ 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.

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.

Where the clast-poor diamictite (Ki1.1.1.3) is narrow or absent, mineralisation occurs within the basal portion of the KPS. This has been modelled as the KPS SMZ, and is best developed around the edges of the Kamoa Dome in the Kansoko Nord, Kamoa Ouest and Kamoa North areas. The Bonanza Zone is also hosted within the KPS, but is modelled as a separate mineralised zone in close proximity to the Bonanza Fault. In the far northern extents at Kamoa North, the Ki1.1.1 and KPS have onlapped onto the R4.2, allowing the Ki1.1.3 direct contact with the R4.2. A separate Ki1.1.3 mineralised zone is modelled in these areas.

The assay file, with the SMZ selections flagged, was then imported into Datamine mining software where it was combined with the collar and survey files. The SMZ selection fields were added to the de-surveyed drillhole file.

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.

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 honour both the vertical and lateral controls on mineralisation. Eleven domains are modelled (Figure 14.1). These were applied to 1 m composite drillholes and the block model. Contacts between domains are treated as hard contacts for resource estimation purposes.

Figure 14.1 Schematic Illustrating the Vertical Position of the Estimation Domains (Localised Domain 50 and Domain 60 in the Far North Excluded)

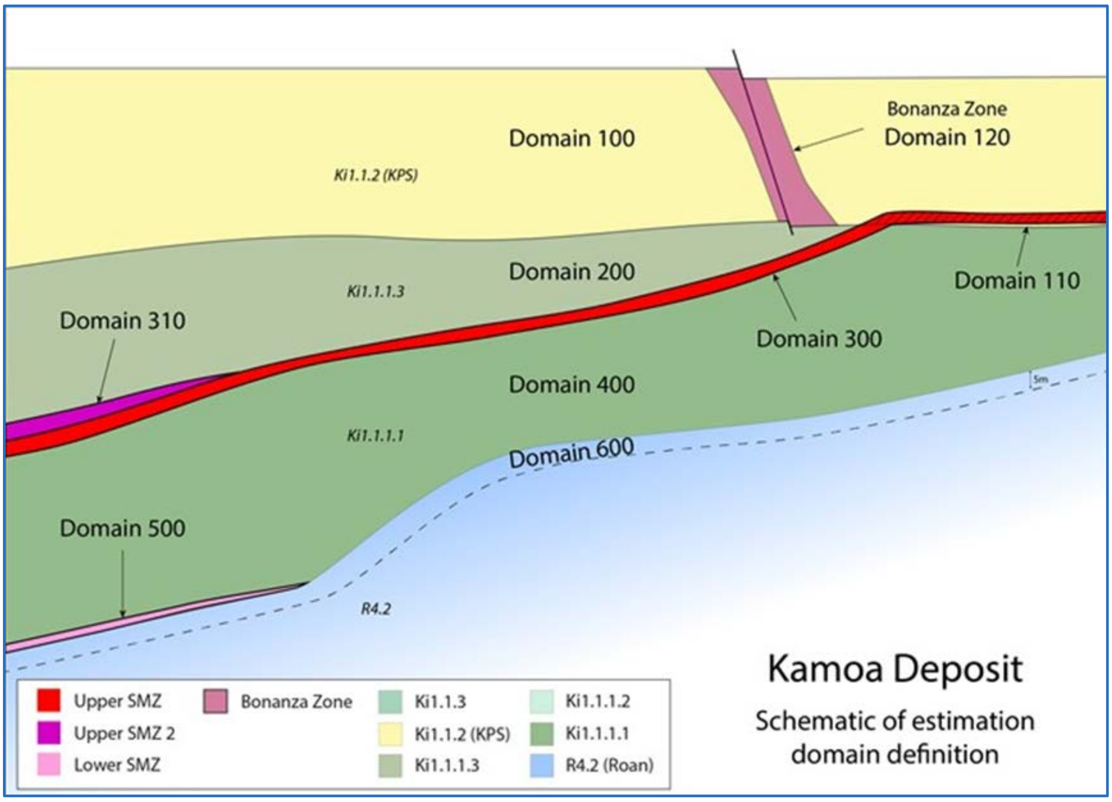


Figure provided by Ivanhoe, 2020.

14.3.2 Kakula

At Kakula, the individual lithological units were combined with the mineralised zone to form six domains used for resource estimation (Figure 14.2). These were applied to 1 m composite drillholes and 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.

Figure 14.2 Kakula: Vertical Domain Definition

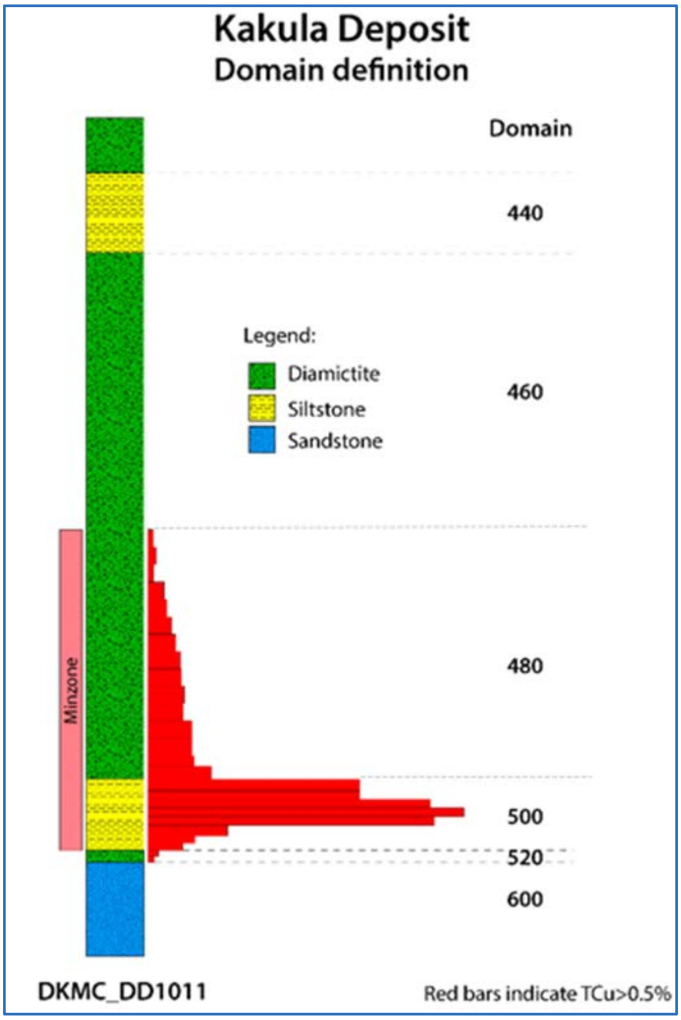


Figure provided by Ivanhoe, 2019.

14.4 Top Capping

14.4.1 Kamoa

Drillhole samples were first combined into 1 m composites, honouring the domain contacts, and then capped. Capping was based on a combination of histogram and log probability plot analysis, review of coefficients of variation (CV), and spatial analysis of higher-grade samples. A lower capping threshold (as a proportion of the distribution) was applied to domains with limited data. The highest grades are typically clustered and show good connectivity between drillholes. As a result, they were either not capped, or had a light capping applied. Top capping values were applied per domain, where necessary, prior to estimation (Table 14.1).

The Kamoa North Bonanza Zone (Domain 120) represents a unique mineralising event at Kamoa, where the controlling east-west growth fault structure allowed oxidised, copper-rich brines to bypass the lower redox interface at the Roan-Nguba contact and instead accessed the overlying, highly-sulfidic and reduced KPS. The new, upper mineralised zone hosted in the KPS is characterised by very high grades, frequently in excess of 20% TCu. Top capping values were applied, but at a high threshold given the continuity of high grades in this domain.

Table 14.1 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	19,927	3.0%	12	0.05	3.86	0.05	2.51
110	653	15.0%	4	2.51	0.94	2.50	0.92
120	609	35.0%	6	7.90	0.86	7.87	0.85
200	8,623	2.6%	14	0.22	1.44	0.22	1.42
300	4,627	18.0%	6	2.70	0.82	2.69	0.81
400	11,240	2.5%	8	0.34	0.89	0.34	0.85

14.4.2 Kakula

At Kakula, top capping was evaluated using 1 m composites within the mineralised zone to assess if isolated high-grade 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 aligns 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 14.3). 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 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 14.2.

Figure 14.3 Kakula: Visual Top Capping Analyses with TCu grades >8%, >10%, 12%, and >14%

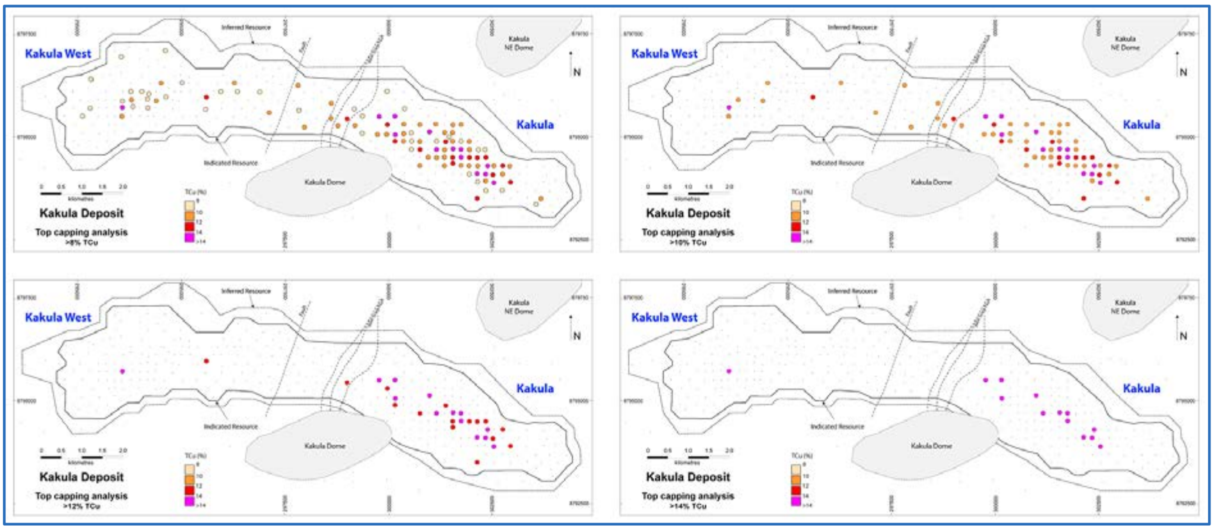


Figure provided by Ivanhoe, 2019.

Table 14.2 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	6,574	1.5	11	0.11	1.88	0.11	1.73
480	3,976	8.0	2	1.25	0.80	1.25	0.80
520	1,269	3.0	6	0.47	1.11	0.46	0.92
600	1,648	2.0	4	0.07	2.98	0.06	2.54

14.5 Exploratory Data Analysis (EDA)

14.5.1 Kamoa

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. Histograms and log probability plots for the Kamoa North Bonanza Zone (Domain 120) and the Upper SMZ (Domain 300)are displayed in Figure 14.4.

Figure 14.4 Kamoa: Histograms of 1 m Composites for TCu (%) for Domains 120 (top) and 300 (bottom)

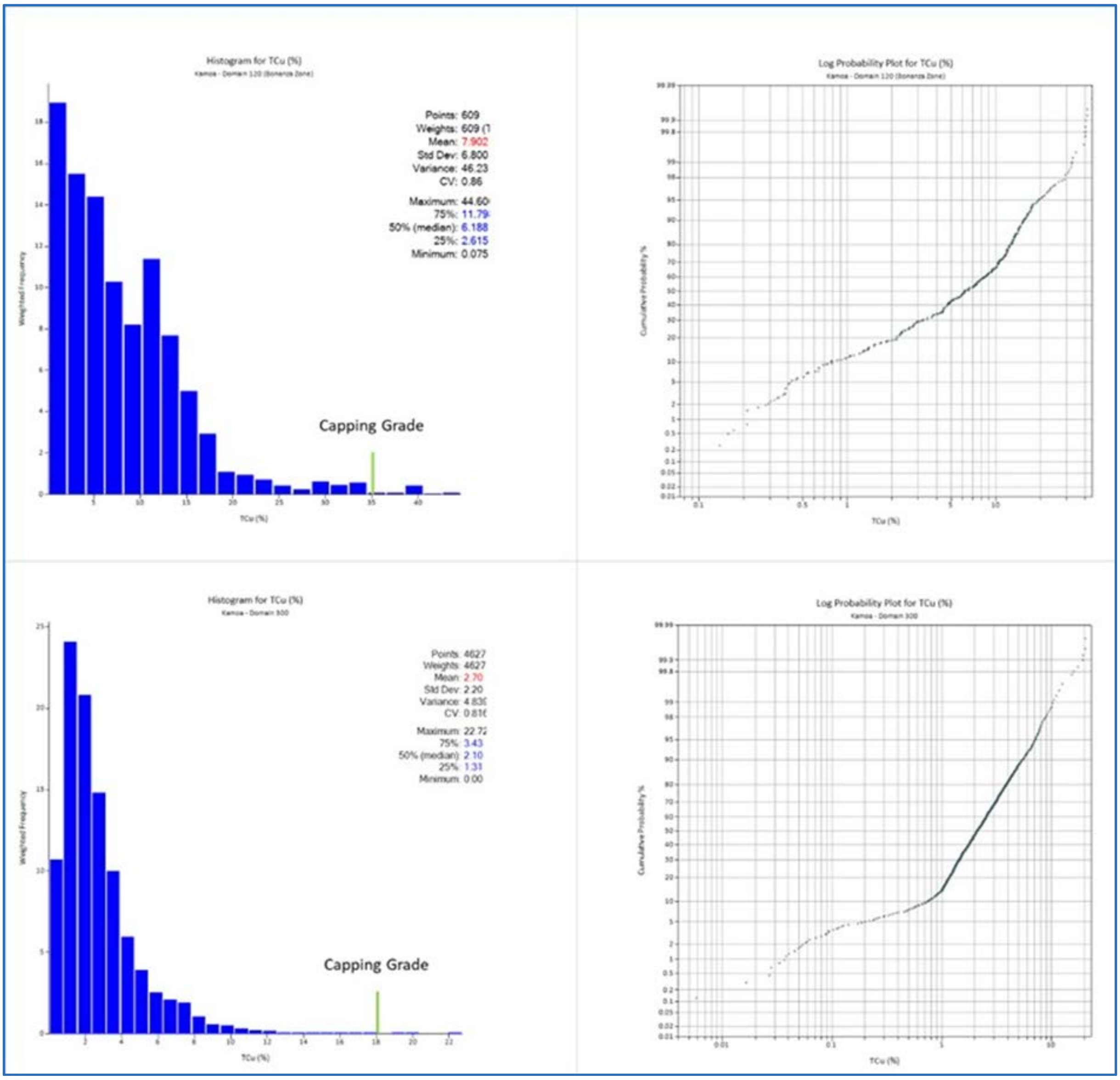


Figure provided by Ivanhoe, 2020. Green lines represent the top capping applied per domain.

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 (Domains 100, 110 and 120) 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 2.57 to 2.69. The Upper SMZ (Domain 300) has a SG of 2.67, towards the upper end of the diamictite range, likely due to the denser sulfide mineralisation. The porous R4.2 sandstone (Domain 600) has the lowest average SG of 2.48. The SG CV for individual domains is low, typically 0.1 or lower.

Sulfur grades are elevated in the KPS due to high concentrations of pyrite within the siltstone. Sulfur grades are also elevated in the mineralised domains, where chalcopyrite dominates. A variety of sulfide species occur within Domain 300 with bornite and chalcocite lowering the overall sulfur grade. Domain 500 is chalcocite-dominant; hence the lower sulfur grades. Overall, sulfur values are positively skewed.

Arsenic values at Kamoa are very low, with approximately 65% of samples <0.001% As.

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 sulfide and oxide mineralisation into separate domains. Geological and metallurgical studies of the sulfide species indicate that the bulk of the mineralisation at Kamoa is sulfide, with localised oxide mineralisation closer to surface and along the edges of the domes. In general, most samples have an ASCu:TCu ratio of 10% or less (representative of sulfides where a small amount will dissolve in sulfuric acid), and very few have a ratio of over 30%, which would typically require selection of reagents that would coat the copper oxide minerals to make them float.

14.5.2 Kakula

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 (Figure 14.5). The bimodality is accounted for in the resource estimation by aligning the anisotropy of the search ranges and variography with the trends of the high-grade material.

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, and used to assign an SG value to those samples with missing SG values.

Figure 14.5 Kakula: 1 m Composite TCu (%) for the mineralised diamictite (Domain 480) and the mineralised basal siltstone (Domain 500). Histogram and Probability Plot

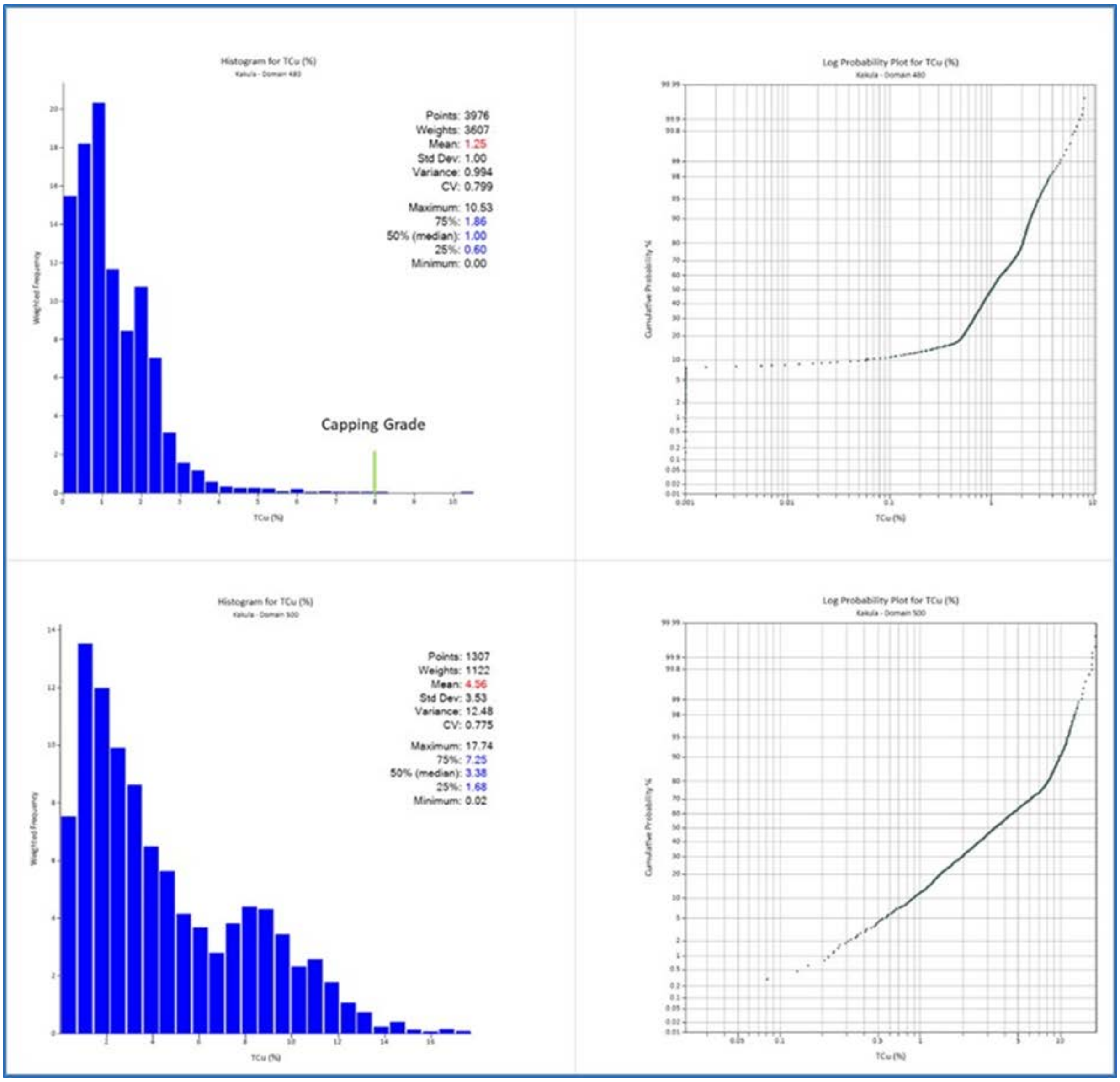


Figure provided by Ivanhoe, 2020.

14.6 Structural Model

14.6.1 Kamoa

Four structures were defined at Kamoa using geophysical data, and lithological discontinuities interpreted from the drillhole data. These structures were then used to divide the model into five structural zones. For grade estimations, the blocks and drillholes were transformed to dilated space, with the SMZs allowed to be included across the structural domain boundaries in the estimation.

Currently, it is difficult to establish the dips of the interpreted faults, and/or 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.

14.6.2 Kakula

Five structural blocks were defined at Kakula. 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 mineralisation and its implication on the efficacy of the proposed grade control and mining methods.

14.7 Surface and Block Modelling

14.7.1 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 five structural domains based on the structural model and coded with grade domains using wireframes that define the stratigraphic units and mineralised zones. A prototype model was established using 50 m x 50 m blocks in easting and northing, with 1 m blocks in elevation. Tighter drillhole spacing, and wireframe geometry were required to outline the narrow Bonanza Zone and a prototype model with 5 m x 5 m x 1 m blocks was used. The two models at different block sizes are mutually exclusive.

14.7.2 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 prototype model was established using 50 m x 50 m blocks in easting and northing, with 1 m blocks in elevation.

14.8 Grade Estimation

14.8.1 Kamoa

To improve stationarity for grade estimation, both the drillholes and the block model were transformed ("dilated") to ensure that the vertical TCu grade profiles match between drillholes. Typically, these profiles are bottom-loaded, with the higher-grades occurring at the bottom of the profile and grading upwards to lower grades towards the top of the profile. The transformation was performed by adjusting the Z-coordinate of the data to 'dilate' the 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 14.6). The exception to this was for the Bonanza Zone (Domain 120), where structural controls, rather than stratiform sedimentary controls, dominate. No transformation was performed for this domain.

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 performed in transformed space. The block models were then transformed back to their original vertical location by setting the centroid of each block back to its original Z-coordinate.

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 a normal-score transform prior to modelling of the variogram. The vertical bandwidth was a key parameter to preserve the vertical TCu drillholes grade profiles, and was typically set to a narrow interval. Downhole variograms of the transformed 1 m composites were used to determine the nugget (C_0). The transformation expands 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. No elevation transform was applied to the Bonanza Zone, and variograms were modelled from samples in their true coordinate positions. Example TCu variograms for Domain 120 (Bonanza Zone) and Domain 300 (Upper SMZ) are shown in Figure 14.7 and Figure 14.8.

Figure 14.6 Kamoa: Vertical Section Showing Untransformed Composites and Blocks (Top) and Transformed Composites and Blocks (Lower) for Domain 300, 3 x Vertical Exaggeration

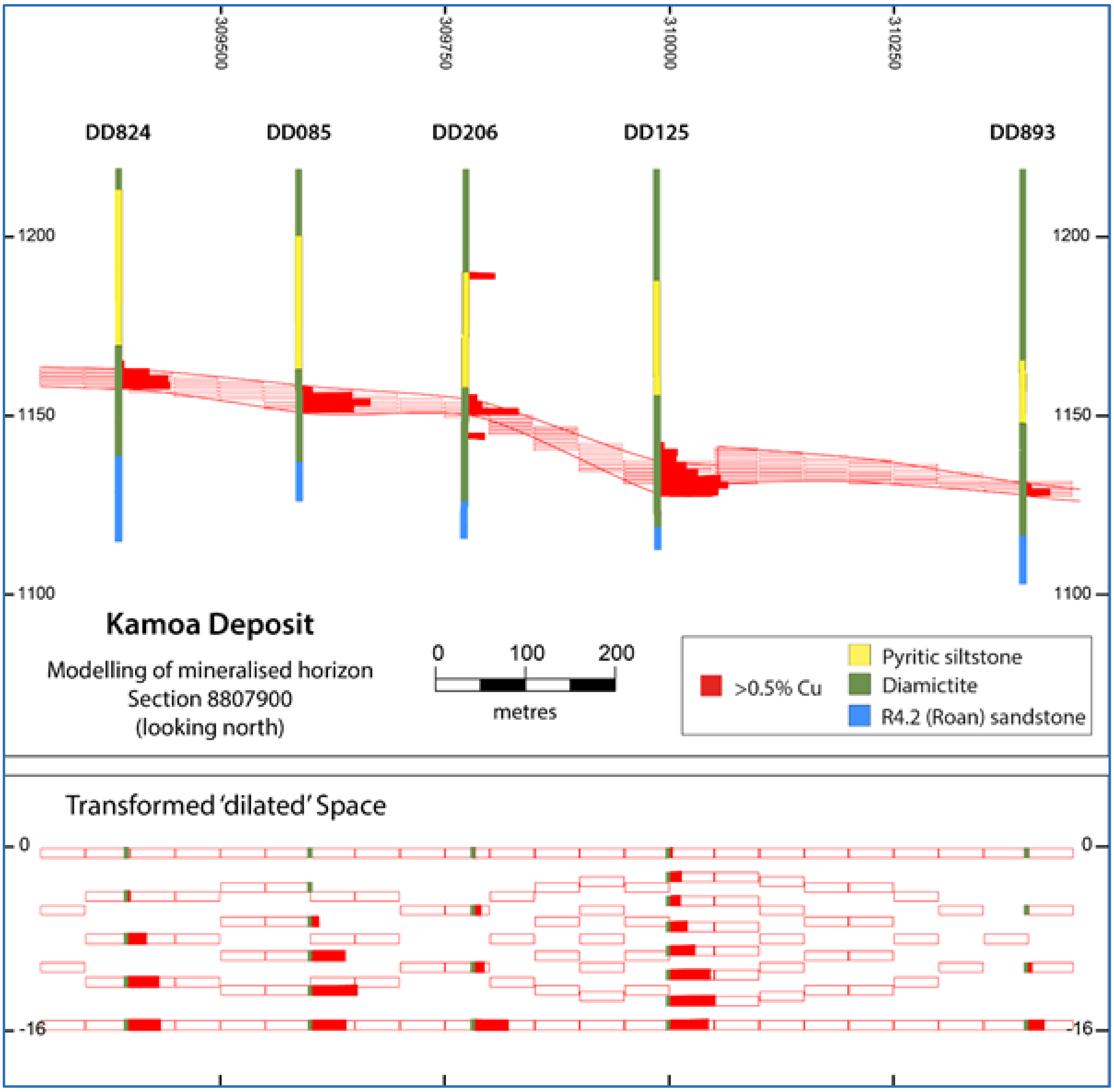


Figure prepared by Ivanhoe, 2019; Copper grade intensity shown by bars on right side of hole.

Figure 14.7 Kamoa: Normal Score Major and Semi-Major Direction Variograms for TCu (Domain 120)

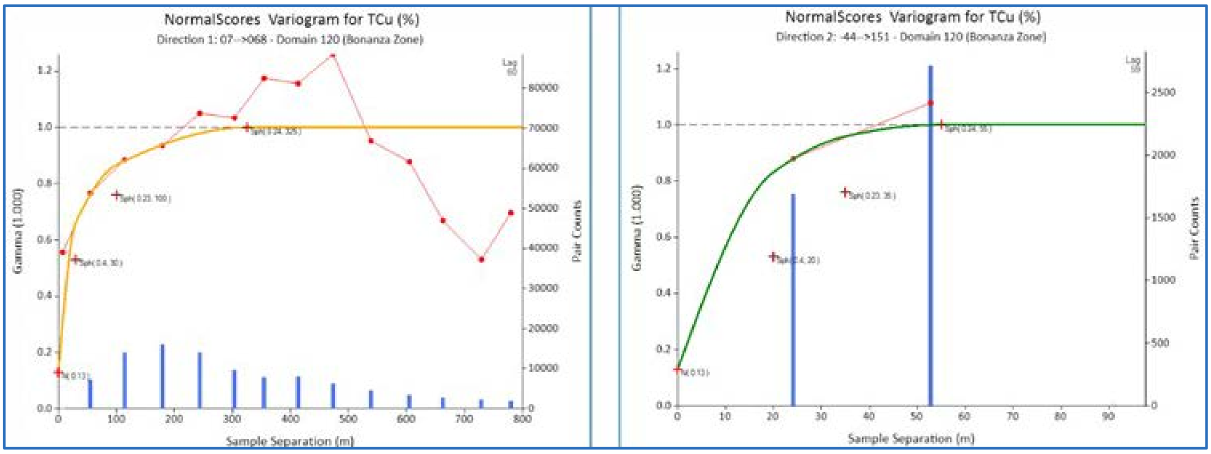


Figure prepared by Ivanhoe, 2020.

Figure 14.8 Kamoa: Normal Score Major and Semi-Major Direction Variograms for TCu (Domain 300)

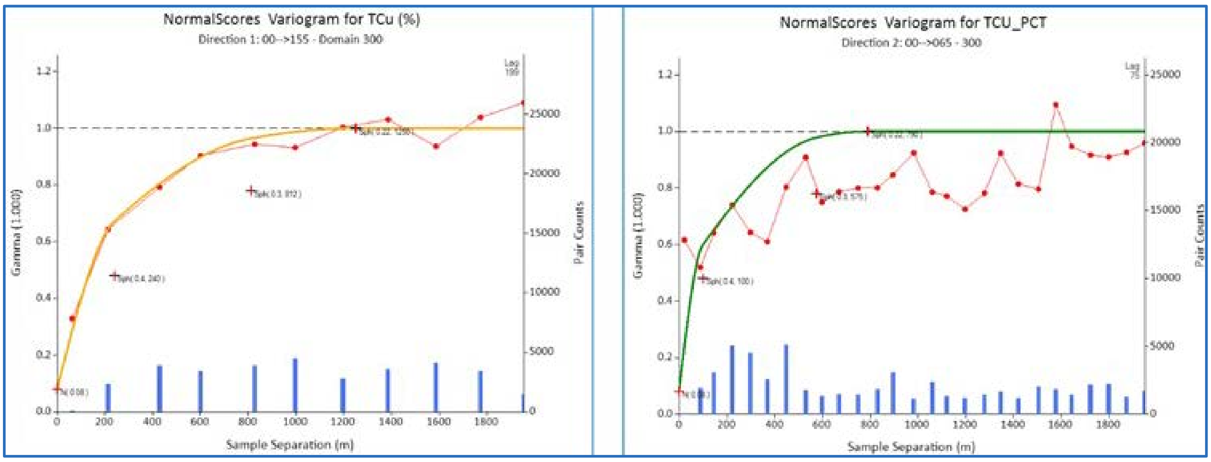


Figure prepared by Ivanhoe, 2020.

All grade variables (TCu, ASCu, As, Fe, and S) were estimated into each block using ordinary kriging (OK) interpolation. The estimated OK grades are used for reporting. In addition, inverse distance to the second power (ID2) and nearest neighbour (NN) estimates were constructed, but were only used for validation purposes. Estimation parameters are summarised in Table 14.3. 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.

Search orientations are not fixed; they vary from block-to-block based upon strike directions of the modelled growth faults that are estimated into the block model (refer to Section 7.3.2) using a Datamine process known as dynamic anisotropy.

Table 14.3 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	160°	0°	1,000	4	12	4	8
	Y	70°	0°	600	4	12	4	8
	Z	0°	90°	5	4	12	4	8
120	X	248°	7°	160	4	12	4	8
	Y	151°	44°	60	4	12	4	8
	Z	345°	45°	20	4	12	4	8
300	X	155°	0°	1,250	4	12	4	8
	Y	65°	0°	600	4	12	4	8
	Z	0°	90°	8	4	12	4	8
310	X	140°	0°	500	4	12	4	8
	Y	50°	0°	250	4	12	4	8
	Z	0°	90°	5	4	12	4	8
500	X	160°	0°	1,000	4	12	4	8
	Y	70°	0°	1,000	4	12	4	8
	Z	0°	90°	20	4	12	4	8

Note: Orientations shown are for overall variography; trend; these trends will vary locally as they follow the variable search orientations based on dynamic anisotropy.

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. This was to prevent 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 29% of TCu samples have a corresponding ASCu value. Within the Upper SMZ (Domain 300), 92% 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 ASCu:TCu ratio was calculated. The final ASCu grade was then back-calculated from the TCu estimate (using all available TCu samples) and the estimated ratio.

Estimated TCu grades for Kamoa are shown in Figure 14.9. A section view through the Bonanza Zone with estimated TCu grades is shown in Figure 14.10.

Figure 14.9 Plan View of Estimated TCu Grades at Kamoa (at a 2% TCu Cut-off)

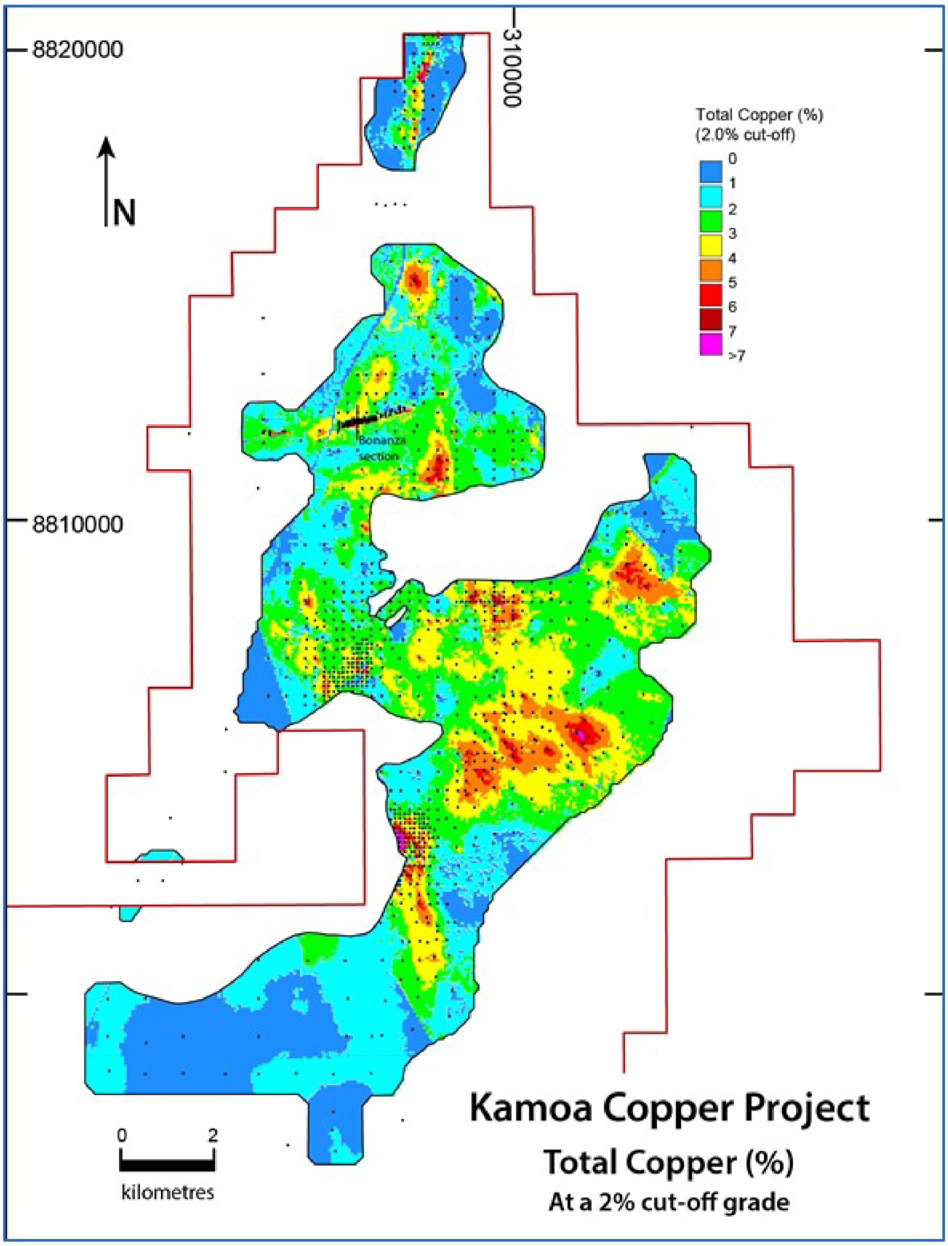


Figure prepared by Ivanhoe, 2020. Image shows the average grade of each vertical stack of blocks above a 2% TCu reporting cut-off.

Figure 14.10 Section View of Estimated TCu Grades in the Bonanza Zone

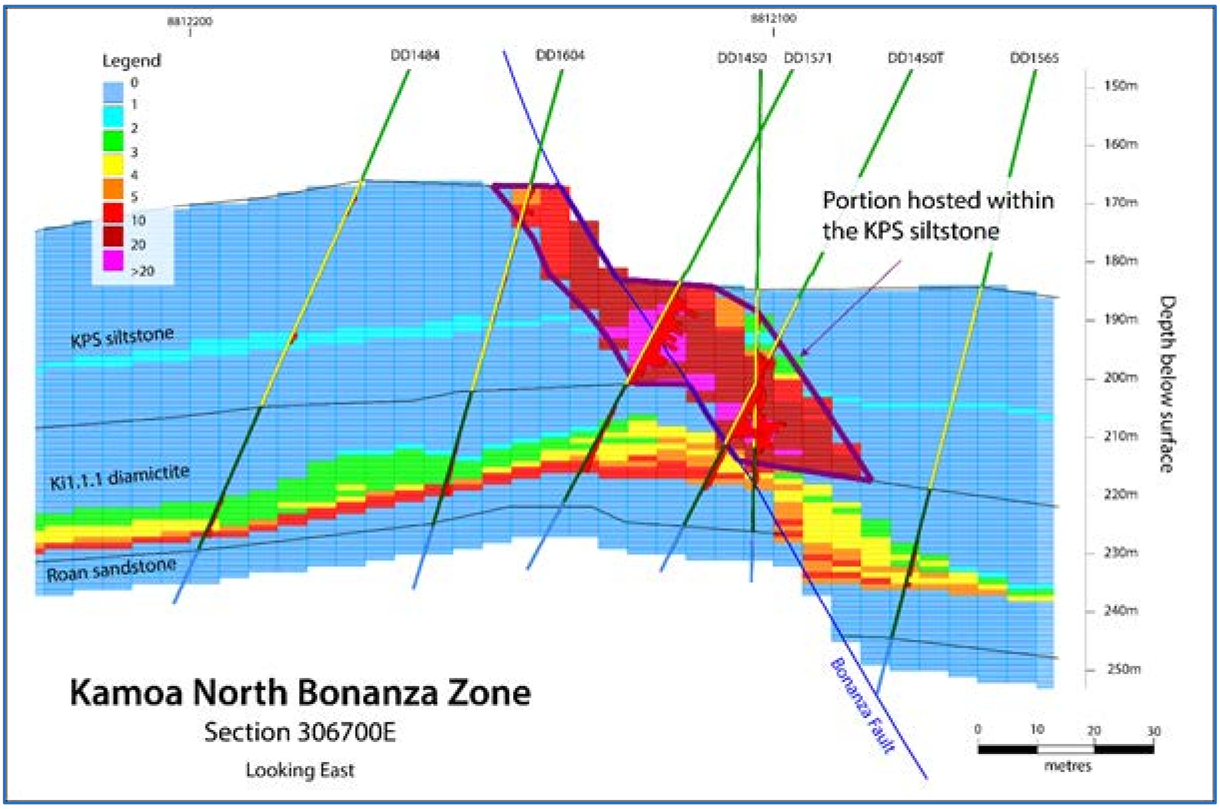


Figure prepared by Ivanhoe, 2020.

14.8.2 Kakula

The same dilational transform estimation method used at Kamoa was applied per domain at Kakula to preserve the strongly-developed bottom-loaded vertical grade profiles that are observed between drillholes. As with Kamoa, hard boundaries were used for individual stratigraphic and mineralisation domains, and soft boundaries were used for structural domains. Variography and estimation were completed in transformed space using 1 m composites. Example TCu variograms for Domain 500 (mineralised basal siltstone) are shown in Figure 14.11.

All grade variables (TCu, As, Fe, and S) were estimated into each block using OK interpolation, and the estimated OK grades were used for reporting. Estimations using ID2 and NN methods were also performed, but only used for validation. ASCu has not been assayed at Kakula and was not included in the estimate. Estimation parameters are summarised in Table 14.4.

Figure 14.11 Kakula: Major and Semi Major Direction Variograms for TCu (Domain 500)

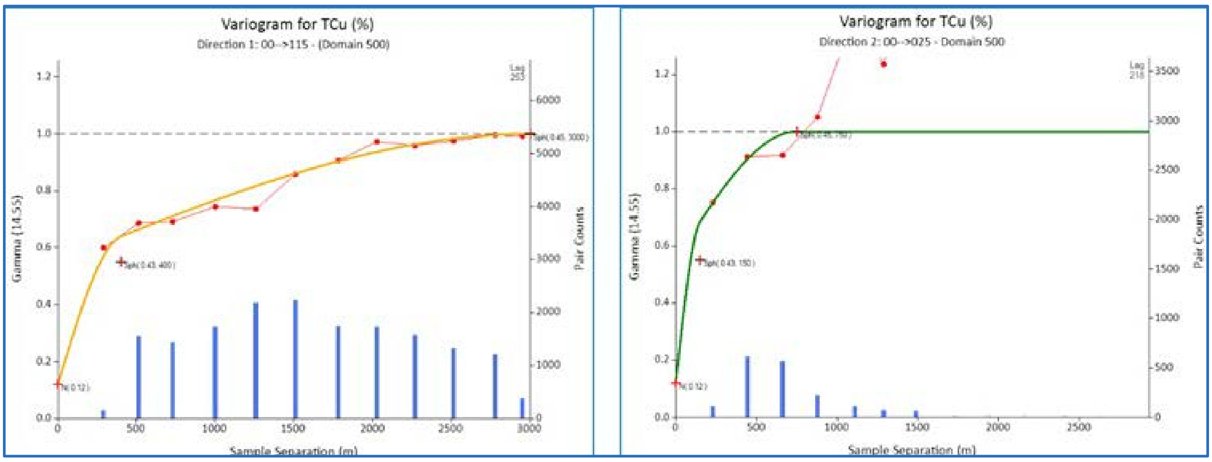


Figure prepared by Ivanhoe, 2020.

Table 14.4 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	075°	0°	800	4	12	OK
	Y	165°	0°	400	4	12	OK
	Z	0°	-90°	5	4	12	OK

NOTE:Orientations shown are for overall search and variography trends; these trends will vary locally as they follow the variable search orientations based on dynamic anisotropy.

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. 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 14.4, Figure 14.2 and Figure 14.12).

Estimated TCu grades for Kakula are shown in Figure 14.13.

Figure 14.12 Kakula: Anisotropy Angles Used at Kakula West, Overlain on the Ki1.1.1-R4.2 (Roan) Contact

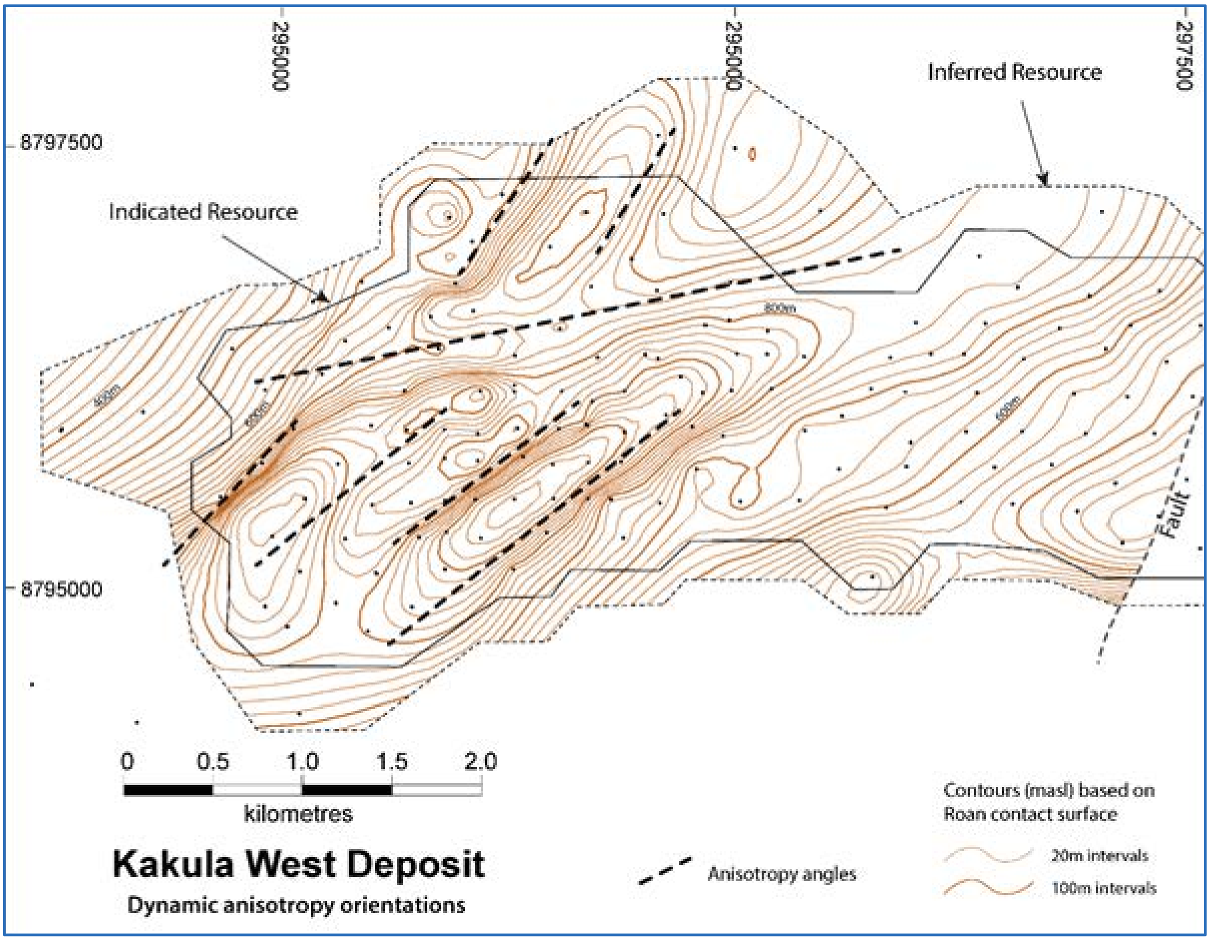


Figure prepared by Ivanhoe 2019.

Figure 14.13 Plan View of Estimated TCu Grades at Kakula

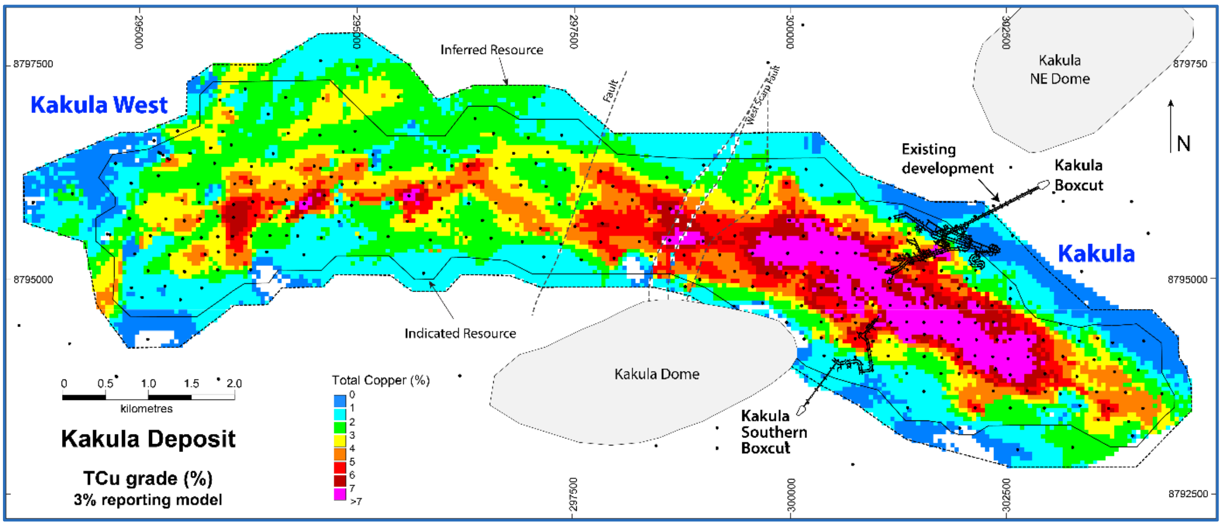


Figure prepared by Ivanhoe 2020. Existing underground development as at September 2020

14.9 Specific Gravity

14.9.1 Kamoa

SG was estimated in transformed space using ID2, using only those SG samples that occurred within individual domain wireframes. Search parameters were the same as those used for sulfur.

14.9.2 Kakula

SG data were available for the majority of drillhole samples, and regression values were used when SG data were missing. SG was estimated as a separate variable, using OK with its own search and variogram parameters.

14.10 Mineral Resource Classification

A number of factors are considered in determining the Mineral Resource classification, including:

- Data quality.
- Drillhole spacing for Inferred Resource and Indicated Resource classification at various comparable stratiform copper mines, particularly in Zambia.
- Variability in elevation and grade between existing drillholes at Kamoa-Kakula over a variety of drillhole spacings from 50 m to 1,600 m.
- Predictability of stratigraphic thicknesses, elevation and grade for new drillholes based on existing models.
- Modelled continuity of mineralisation and robustness of variograms for different domains; modelled continuity ranges far exceed current drillhole spacings used for classification.

- Comparison of modelled geology units and actual underground exposure.

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 27.4 km^2. Mineral Resources within a combined area of 77.0 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 Bonanza Zone represents mineralisation hosted in a more geologically complex environment than is typically the case at Kamoa-Kakula. Two drill sections have been completed in the deeper Bonanza Zone areas (west of the West Scarp Fault) and have been classified as Inferred Mineral Resources. A total of 25 drill sections in the shallower Bonanza Zone mineralisation east of the West Scarp Fault were drilled, spaced 100 m apart along strike, with the central areas drilled on 50 m strike sections. Drillholes are spaced approximately 25-30 m apart on dip. The significantly denser drilling was planned to better define the mineralisation and account for the additional geological complexity, allowing this zone to be classified as Indicated Mineral Resources.

The Mineral Resource classification with drillholes for Kamoa is shown in Figure 14.14, and for Kakula in Figure 14.15.

Figure 14.14 Kamoa: Mineral Resource Classification

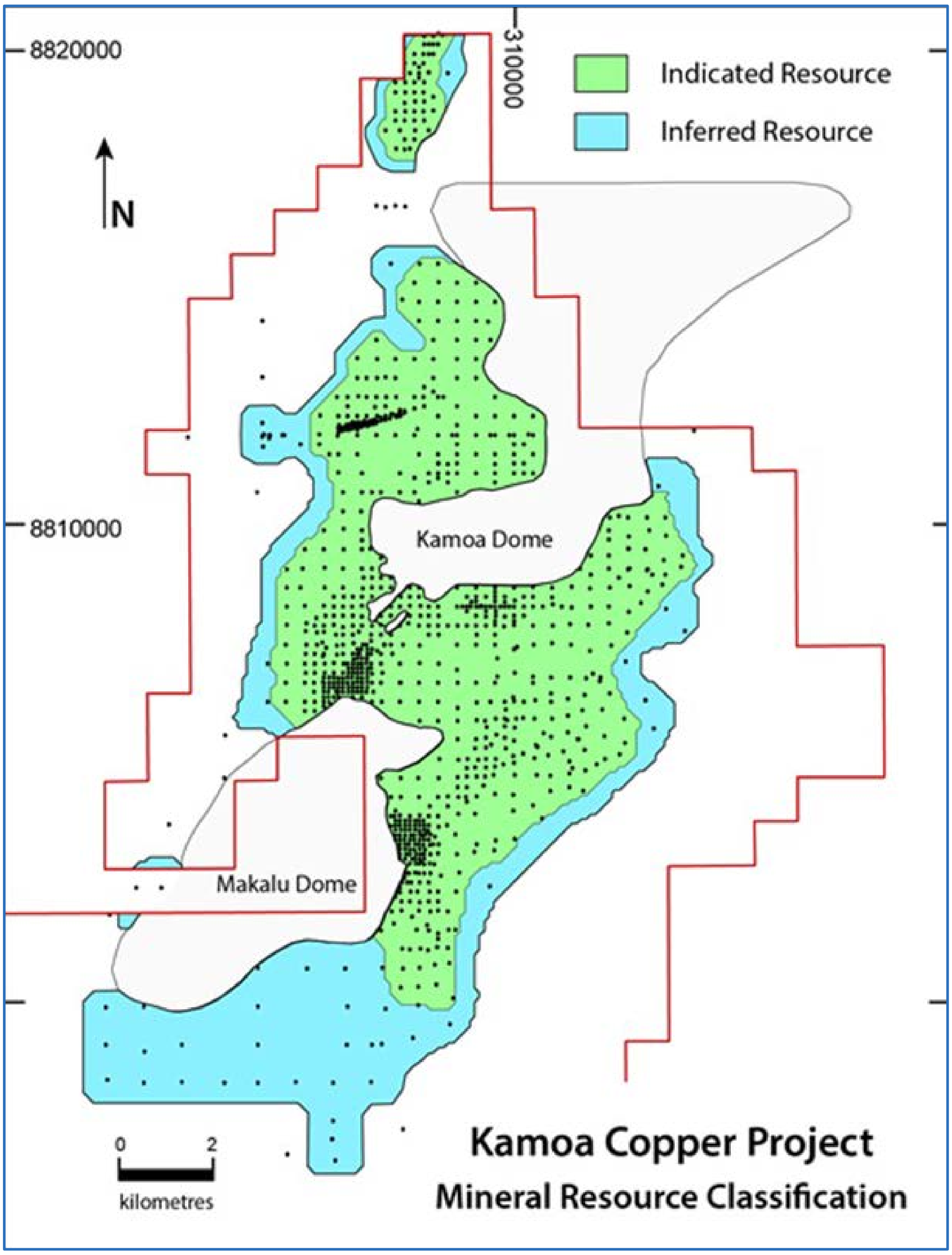


Figure prepared by Ivanhoe, 2020

Figure 14.15 Kakula: Mineral Resource Classification

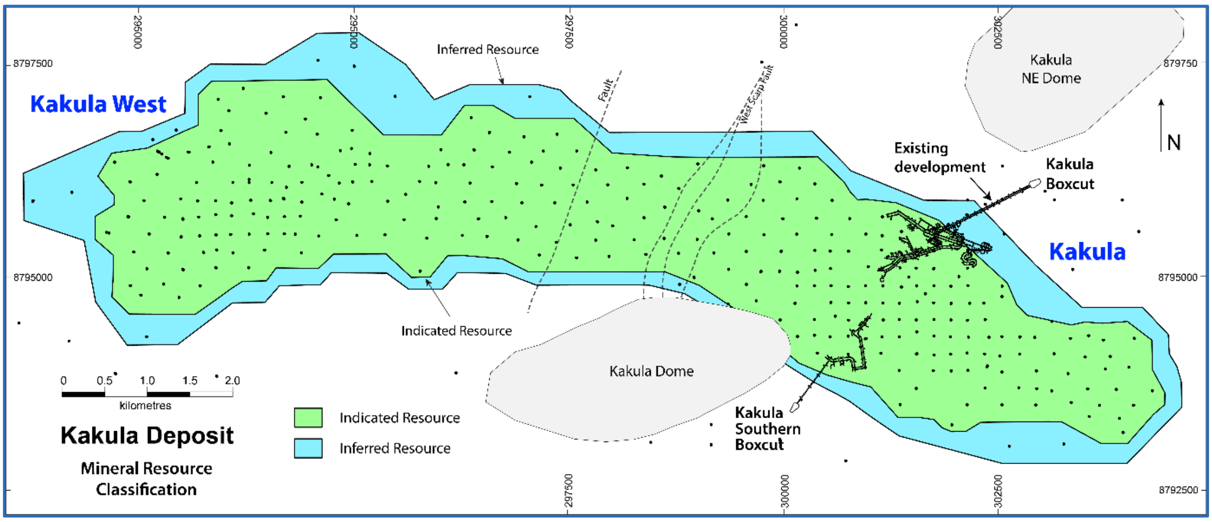


Figure provided by Ivanhoe, 2020. Existing underground development as at September 2020.

14.11 Model Validations

Models were validated using a number of checks:

- Visual checks: estimated block grades and composite grades were compared visually in plan and cross sectional views and showed a good agreement.
- Results from the previous 2D modelling method were compared to results from the current 3D modelling method visually and graphically through plotting the grade and tonnage at different cut-offs, and by plotting the grade distributions in plan view. The SMZ selection and ID2 estimation using an isotropic search in the 2D model have both contributed to a smoothing of the block grades when TCu was estimated using a single composite and single block with a variable height over the entire SMZ interval. In the 3D model, however, this smoothing is minimised when the vertical grade profile is preserved using 1 m composites, 1 m high blocks, and estimations performed in transformed space. The use of an anisotropic search and dynamic anisotropy in the 3D model has also contributed to greater continuity in grade.
- Global bias: nearest neighbour (NN) estimates were used to check for global bias between the estimated grade and the drillhole grades. Relative differences between the ID and NN models are generally below 5%, which is considered appropriate for an Indicated Resource classification.
- Local bias checks: At Kamoa, swath plots were constructed for TCu on 400 m slices (swaths) in easting and northing. No local biases were evident.
- At Kakula, swath plots were constructed for TCu and S on 500 m swaths aligned northwest–southeast (along the trend of the high-grade mineralisation), and 500 m swaths aligned southwest–northeast (across the trend of the high-grade mineralisation). No local biases were evident.

14.12 Reasonable Prospects of Eventual Economic Extraction

Wood used a 1% TCu cut-off grade and a minimum three metre vertical height 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.

14.12.1 Kamoa

To test the cut-off grade for the purposes of assessing reasonable prospects of eventual economic extraction, Wood 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 sulfide copper is estimated to be (TCu-ASCu) + 0.125 TCu. The hypogene recovery equation is then applied to sulfide 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 general and administrative (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.

Frequently, 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 mine throughput rates were increased. These additional areas are included using a 1% TCu cut-off. There is a small percentage (~10%) of the tonnage representing 5% 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 $22/t of $27/t assumed mining costs. It may be convenient to mine these blocks in conjunction with adjacent higher-grade blocks, and therefore Wood 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, Wood 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, sulfuric 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. 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 (0.8% of the tonnage and 0.4% of the metal) of blocks that will not cover full mining costs. These blocks will cover $17/t of $27/t assumed mining costs.

Wood 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.

14.12.2 Kakula

Wood assessed reasonable prospects for eventual economic extraction for Kakula. 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 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, Wood 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 Wood 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 was 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 sulfuric 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 estimate reported at a cut-off grade >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.

14.13 Mineral Resource Statement

Ivanhoe's Vice President, Resources 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 Gordon Seibel, a Registered Member (RM) of the Society for Mining, Metallurgy and Exploration (SME), a Wood employee, who is the Qualified Person for the Mineral Resource estimate

The Mineral Resources were classified in accordance with the 2014 CIM Definition Standards. Mineral Resources are stated in terms of TCu, and a minimum vertical thickness of 3 m.

To avoid reporting isolated blocks above cut-off in both the Kamoa and Kakula models, a minimum vertical stack of three contiguous one metre high 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.

14.13.1 Kamoa

Indicated and Inferred Mineral Resources for the 3D resource model are summarised in Table 14.5. 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 30 January 2020, and the cut-off date for the drill data is 20 January 2020. The Mineral Resources do not include any material in the hanging wall and footwall, and make no allowance for mining recovery factors.

Table 14.5 Kamoa Indicated and Inferred Mineral Resource (at 1% TCu Cut-off Grade)

Category	Tonnage (Mt)	Area (km^2)	Copper (%)	Vertical Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
Indicated	760	55.2	2.73	5.0	20,800	45.8
Inferred	235	21.8	1.70	4.0	4,010	8.8

1. Ivanhoe's Vice President, Resources 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 Gordon Seibel, a Registered Member (RM) of the Society for Mining, Metallurgy and Exploration (SME), employee of Wood, who is the Qualified Person for the Mineral Resource estimate. The effective date of the estimate is 30 January 2020 and the cut-off date for drill data is 20 January 2020. Mineral Resources are reported using the CIM 2014 Definition Standards for Mineral Resources and Mineral Reserves. Mineral Resources 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.
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 hanging wall 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.

14.13.2 Kakula

The Mineral Resources were estimated as of 10 November 2018 and the cut-off date for the drill data is 1 November 2018. On 10 February 2020, the inputs used in assessing reasonable prospects of eventual extraction and the drill data inputs were reviewed to ensure the estimate remained current. There are no changes to the estimate as a result of the review, and the estimate has an effective date of 10 February 2020.

The Kakula Mineral Resource is summarised in Table 14.6 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 hanging wall and footwall dilution skins and make no allowance for mining recovery factors.

Table 14.6 Kakula: Indicated and Inferred Mineral Resource (at 1% TCu Cut-off Grade)

Category	Tonnage (Mt)	Area (km^2)	Copper (%)	Vertical Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
Indicated	627	21.7	2.74	10.3	17,200	37.9
Inferred	104	5.6	1.61	6.7	1,680	3.7

1. Ivanhoe's Vice President, Resources 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 Gordon Seibel, a Registered Member (RM) of the Society for Mining, Metallurgy and Exploration (SME), employee of Wood, who is the Qualified Person for the Mineral Resources. The Mineral Resources were estimated as of 10 November 2018 and the cut-off date for the drill data is 1 November 2018. On 10 February 2020, the inputs used in assessing reasonable prospects of eventual extraction and the drill data inputs were reviewed to ensure the estimate remained current. There are no changes to the estimate as a result of the review, and the estimate has an effective date of 10 February 2020. Mineral Resources are reported using the CIM 2014 Definition Standards for Mineral Resources and Mineral Reserves. Mineral Resources 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.
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. Depth of mineralisation below the surface ranges from 12–1,373 m for Indicated Mineral Resources and 61–1,397 m for Inferred Mineral Resources.
4. Reported Mineral Resources contain no allowances for hanging wall 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 differences between tonnes, grade and contained metal content.

14.13.3 Kamoa–Kakula Project

Indicated and Inferred Mineral Resources for the Kamoa–Kakula Project are provided on a 100% basis in Table 14.7. The Mineral Resources in Table 14.5 and Table 14.6 are not additive to this table.

Table 14.7 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	760	55.2	2.73	5.0	20,800	45.8
	Inferred	235	21.8	1.70	4.0	4,010	8.8
Kakula	Indicated	627	21.7	2.74	10.3	17,200	37.9
	Inferred	104	5.6	1.61	6.7	1,680	3.7
Total Kamoa-Kakula Project	Indicated	1,387	77.0	2.74	6.5	38,000	83.7
	Inferred	339	27.4	1.68	4.5	5,690	12.5

1. Ivanhoe's Vice President, Resources, 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 Gordon Seibel, a Registered Member (RM) of the Society for Mining, Metallurgy and Exploration (SME), who is the Qualified Person for the Mineral Resource estimate. The effective date of the estimate for Kamoa is 30 January 2020, and the cut-off date for drill data is 20 January 2020. The Mineral Resources were estimated as of 10 November 2018 and the cut-off date for the drill data is 1 November 2018. On 10 February 2020, the inputs used in assessing reasonable prospects of eventual extraction and the drill data inputs were reviewed to ensure the estimate remained current. There are no changes to the estimate as a result of the review, and the estimate has an effective date of 10 February 2020. Mineral Resources are reported using the CIM 2014 Definition Standards for Mineral Resources and Mineral Reserves. Mineral Resources 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.
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 hanging wall 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 reported in Table 14.5 and Table 14.6 are not additive to this table.

14.14 Sensitivity of Mineral Resources to Cut-off Grade

Table 14.8 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 14.5 and Table 14.7 are not additive to this table.

Table 14.9 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. Mineral Resources reported in Table 14.6, and Table 14.7 are not additive to this table.

Table 14.8 Kamoa: Sensitivity of Mineral Resources to Cut-off Grade

Indicated Mineral Resource						
Cut-off (% Cu)	**Tonnage (Mt)**	**Area (km^2)**	**Copper (%)**	**Vertical Thickness (m)**	**Contained Copper (kt)**	**Contained Copper (billion lbs)**
5.0	44	4.5	6.14	3.5	2,690	5.9
4.5	67	6.7	5.65	3.6	3,800	8.4
4.0	107	10.4	5.13	3.7	5,490	12.1
3.5	171	16.4	4.61	3.7	7,890	17.4
3.0	256	24.0	4.15	3.8	10,700	23.5
2.5	369	32.8	3.73	4.1	13,700	30.3
2.0	504	41.5	3.33	4.4	16,800	37.0
1.5	655	49.4	2.97	4.8	19,400	42.8
1.0	760	55.2	2.73	5.0	20,800	45.8
0.5	1,185	59.4	1.99	7.3	23,600	52.0
Inferred Mineral Resource						
4.0	1	0.1	5.47	3.4	55	0.1
3.5	4	0.5	4.12	3.1	177	0.4
3.0	13	1.5	3.51	3.1	441	1.0
2.5	30	3.5	3.08	3.0	910	2.0
2.0	58	6.5	2.66	3.2	1,540	3.4
1.5	113	11.9	2.20	3.4	2,480	5.5
1.0	235	21.8	1.70	4.0	4,010	8.8
0.5	680	31.4	1.01	8.0	6,860	15.1

Note:The footnotes to Table 14.5 also apply to this table. Mineral Resources reported in Table 14.5 and Table 14.7 are not additive to this table.

Table 14.9 Kakula: Sensitivity of Mineral Resources to Cut-off Grade

Cut-off (% Cu)	Tonnage (Mt)	Area (km²)	Copper (%)	True Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
Indicated Mineral Resource						
5.0	77	5.9	7.48	4.5	5,730	12.6
4.5	91	7.0	7.04	4.5	6,440	14.2
4.0	109	8.3	6.58	4.6	7,200	15.9
3.5	132	9.9	6.09	4.7	8,060	17.8
3.0	167	11.8	5.50	5.0	9,180	20.2
2.5	218	14.3	4.85	5.4	10,600	23.3
2.0	318	17.5	4.02	6.5	12,800	28.2
1.5	435	19.6	3.41	7.9	14,900	32.7
1.0	627	21.7	2.74	10.3	17,200	37.9
0.5	939	22.6	2.08	14.9	19,500	43.0
Inferred Mineral Resource						
4.0	1	0.1	4.41	3.3	33	0.1
3.5	2	0.2	4.04	3.6	67	0.1
3.0	5	0.4	3.52	3.9	168	0.4
2.5	10	1.0	3.10	3.7	324	0.7
2.0	22	2.0	2.64	3.9	583	1.3
1.5	45	3.7	2.18	4.3	974	2.1
1.0	104	5.6	1.61	6.7	1,680	3.7
0.5	257	7.9	1.08	11.7	2,770	6.1

Note:The footnotes to Table 14.6 also apply to this table. This table is not additive to Table 14.6 and Table 14.7.

Table 14.10 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. Mineral Resources reported in Table 14.5 to Table 14.9 are not additive to this table.

Table 14.10 Kamoa and Kakula: Sensitivity of Project Mineral Resources to Cut-off Grade

Cut-off (% Cu)	Tonnage (Mt)	Area (km²)	Copper (%)	Vertical Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
Indicated Mineral Resource						
5.0	120	10.4	6.99	4.1	8,420	18.6
4.5	159	13.7	6.45	4.1	10,200	22.6
4.0	217	18.7	5.86	4.1	12,700	28.0
3.5	304	26.3	5.25	4.1	16,000	35.2
3.0	423	35.8	4.68	4.2	19,900	43.7
2.5	587	47.1	4.14	4.5	24,300	53.6
2.0	823	59.0	3.60	5.0	29,600	65.3
1.5	1,090	69.0	3.15	5.7	34,300	75.6
1.0	1,387	77.0	2.74	6.5	38,000	83.7
0.5	2,123	82.0	2.03	9.4	43,100	95.0
Inferred Mineral Resource						
4.0	2	0.2	5.02	3.4	88	0.2
3.5	6	0.6	4.10	3.2	244	0.5
3.0	17	1.9	3.51	3.2	609	1.3
2.5	40	4.5	3.08	3.2	1,230	2.7
2.0	80	8.5	2.66	3.4	2,120	4.7
1.5	157	15.6	2.19	3.6	3,450	7.6
1.0	339	27.4	1.68	4.5	5,690	12.5
0.5	937	39.3	1.03	8.7	9,630	21.2

Note:The footnotes to Table 14.7 also apply to this table. This table is not additive to Table 14.5, Table 14.6, Table 14.7, Table 14.8, and Table 14.9.

Sensitivity tables within the Kamoa North area are further divided into three areas (Figure 14.16):

- Bonanza Zone, incorporating the elevated grades within both the Ki1.1.1 (Domain 300) and KPS-hosted mineralisation (Domain 120) (Table 14.11).
- The zone of elevated grade associated with the Bonanza Fault (Domain 120), where mineralising fluids have had direct access to the highly reducing KPS (Table 14.12).
- Kamoa Far North, in the furthest northern extent of the Mineral Resource on the mining permit (Table 14.13).

Mineral Resources reported in Table 14.5, Table 14.7, Table 14.8 and Table 14.10 are not additive to these tables.

Figure 14.16 Location of Bonanza Zone and Kamoa Far North within Kamoa North

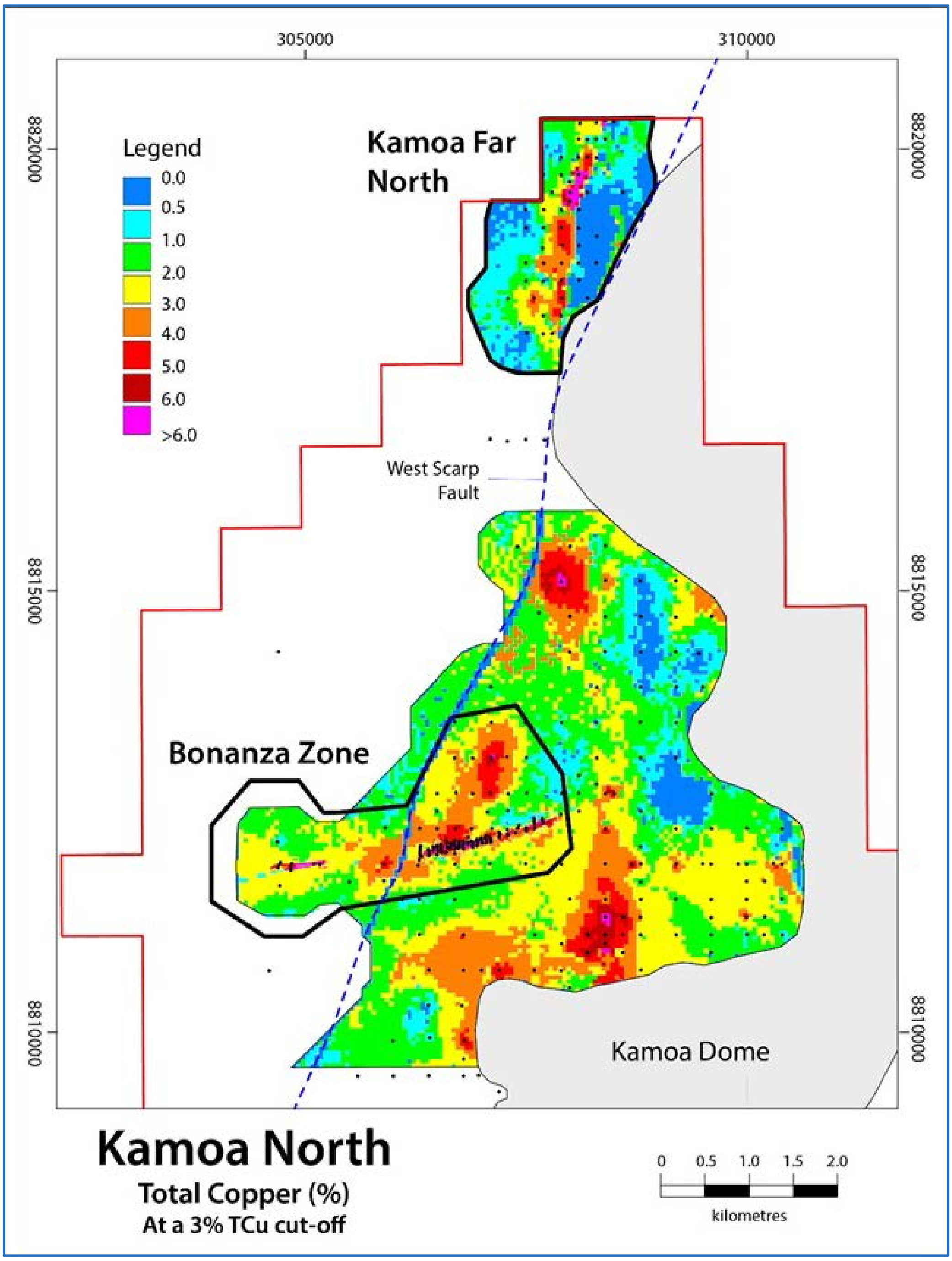


Figure provided by Ivanhoe, 2020. Image shows the average grade of each vertical stack of blocks above a 3% TCu reporting cut-off.

OreWin

Table 14.11 Kamoa Bonanza Zone: Sensitivity of Mineral Resources to Cut-off Grade

Cut-off (% Cu)	Tonnage (Mt)	Area (km^2)	Copper (%)	Vertical Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
Indicated Mineral Resource						
5.0	2	0.1	8.89	6.9	212	0.5
4.5	3	0.2	7.85	5.8	250	0.6
4.0	4	0.3	6.84	5.0	303	0.7
3.5	8	0.7	5.53	4.2	421	0.9
3.0	12	1.1	4.65	4.1	574	1.3
2.5	20	1.8	3.95	4.0	773	1.7
2.0	27	2.4	3.50	4.2	933	2.1
1.5	33	2.8	3.15	4.4	1,050	2.3
1.0	37	3.1	2.95	4.5	1,100	2.4
0.5	49	3.4	2.41	5.4	1,170	2.6
Inferred Mineral Resource						
3.0	1	0.1	5.35	4.1	41	0.1
2.5	2	0.2	3.84	3.4	72	0.2
2.0	9	0.8	2.55	3.8	227	0.5
1.5	16	1.5	2.20	4.1	362	0.8
1.0	19	1.6	2.09	4.3	388	0.9
0.5	55	1.6	1.11	12.2	612	1.3

Note:The footnotes to Table 14.5 also apply to this table. Mineral Resources reported in Table 14.5, Table 14.7, Table 14.8, Table 14.10, Table 14.12 and Table 14.13 are not additive to this table.

OreWin

Table 14.12 Kamoa Bonanza Zone hosted within the KPS (Domain 120): Sensitivity of Mineral Resources to Cut-off Grade

Cut-off (% Cu)	Tonnage (Mt)	Area (km^2)	Copper (%)	Vertical Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
Indicated Mineral Resource						
5.0	1.5	0.06	10.68	10.5	162	0.36
4.5	1.6	0.06	10.52	10.4	165	0.36
4.0	1.6	0.06	10.35	10.4	167	0.37
3.5	1.6	0.06	10.22	10.4	168	0.37
3.0	1.7	0.06	10.11	10.4	169	0.37
2.5	1.7	0.06	9.95	10.3	170	0.37
2.0	1.7	0.07	9.77	10.2	171	0.38
1.5	1.8	0.07	9.68	10.2	171	0.38
1.0	1.8	0.07	9.55	10.1	172	0.38
0.5	1.8	0.07	9.44	10.1	172	0.38
Inferred Mineral Resource						
3.0	0.4	0.03	6.95	4.9	30	0.1
2.5	0.5	0.03	6.74	4.9	30	0.1
2.0	0.5	0.03	6.52	5.0	31	0.1
1.5	0.5	0.03	6.24	5.3	32	0.1
1.0	0.5	0.03	6.24	5.2	32	0.1
0.5	0.5	0.03	6.24	5.2	32	0.1

Note:The footnotes to Table 14.5 also apply to this table. Mineral Resources reported in Table 14.5, Table 14.7, Table 14.8, Table 14.10, Table 14.11 and Table 14.13 are not additive to this table.

Table 14.13 Kamoa Far North: Sensitivity of Mineral Resources to Cut-off Grade

Cut-off (% Cu)	Tonnage (Mt)	Area (km^2)	Copper (%)	Vertical Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
Indicated Mineral Resource						
5.0	1	0.1	7.17	4.0	78	0.2
4.5	1	0.1	6.58	3.9	94	0.2
4.0	2	0.2	5.69	4.0	133	0.3
3.5	3	0.3	5.09	4.0	171	0.4
3.0	5	0.5	4.49	4.0	222	0.5
2.5	7	0.7	3.92	4.2	287	0.6
2.0	11	0.9	3.39	4.5	365	0.8
1.5	15	1.1	2.97	4.9	432	1.0
1.0	18	1.4	2.65	4.7	473	1.0
0.5	24	1.9	2.15	4.7	517	1.1
Inferred Mineral Resource						
1.5	0.2	0.04	1.97	2.7	5	0.0
1.0	2	0.2	1.32	2.9	21	0.0
0.5	6	0.8	0.88	2.9	50	0.1

Note:The footnotes to Table 14.5 also apply to this table. Mineral Resources reported in Table 14.5, Table 14.7, Table 14.8, Table 14.10, Table 14.11 and Table 14.12 are not additive to this table.

14.15 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 historically 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 potentially improve the Project economics.

14.16 Targets for Further Exploration

Specific targets for further exploration are not currently defined at Kamoa-Kakula.

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.

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. There is excellent potential for discovery of additional mineralisation.

14.17 Comments on Section 14

Mineral Resources for the Project have been estimated using core drill data and conform to the requirements of CIM Definition Standards (2014). Wood has checked the data and the methodology used to construct the resource model (Datamine macros) and has validated the resource model. Wood finds the Kamoa and Kakula resource models to be suitable to support feasibility 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.
    - Has yet to be determined for both deposits. Ivanhoe plans to study these risks with the declines currently in progress at Kamoa and Kakula.
    - In the Kakula south developments, minor offsets across growth faults have been encountered, but adjustments to the he mining methods has allowed the mining to follow the steeper dips of the mineralisation across the faults.
    - In the Kakula northern access drive, a larger growth fault was encountered where the mineralisation of the south side of the fault was faulted down (with variable offsets). A spiral decline was developed to accommodate the offsets, and re-established mining on the mineralisation.

- The Kakula southern and northern declines and associated development are expected to join towards the end of 2020. This will provide a complete section across the deposit to further study the structural geology of the Kakula deposit.

- 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 Kakula Mineral Reserve Estimate

The Kakula 2020 FS 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 15.1. The Mineral Reserve is based on the 2019 Mineral Resource. The Mineral Reserve estimate in the 6.0 Mtpa scenario is based on the resource block model provided to Stantec in March 2019 (filename: *Model No mo_kak_nsr21_190628*).

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 2020 FS has not used all the Mineral Resources available to be converted to Mineral Reserve. The mining reserve focused on maximising the grade profile for a 6.0 Mtpa full production rate for 15-years, with emphasis on further maximising grade early in the mine life. Using the modelled 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 15.1 Kakula 2020 Mineral Reserves

Classification	Ore (Mt)	Copper (%)	Copper (Contained Mlb)	Copper (Contained kt)
Proven Mineral Reserve	–	–	–	–
Probable Mineral Reserve	110.0	5.22	12,655	5,745
Mineral Reserve	110.0	5.22	12,655	5,745

1. Effective date of the Kakula Mineral Reserve is 8 September 2020.
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$3.10/lb.
4. An elevated cut-off of US$100/t NSR was used to define the stoping shapes.
5. Indicated Mineral Resources were used to report 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.
8. Rounding as required by reporting guidelines may result in apparent differences between tonnes, grade, and contained metal content.

All production drift dilution tonnes and grades were calculated by interrogating the diluted design shape with the feasibility Mineral Resource block model. In addition, all secondary, tertiary, and second lift production headings that contain paste dilution have a paste dilution component calculated within the schedule to produce final diluted grades. The paste fill tonnage has a zero-grade copper value. 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 21-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.

15.2 Kansoko Mineral Reserve Estimate

The Kansoko 2020 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 15.2. 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 Kansoko in the Kakula-Kansoko 2020 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 15.2 Kansoko 2020 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 8 September 2020.
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 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 Kansoko Mineral Reserve ranges between depths of 60–1,300 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 6.0 Mtpa ore.

Room-and-pillar mining will be used for ore zones from 60–150 m in depth and from 150–250 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. 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 if the parameters vary positively or negatively, then it is likely 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 Kansoko 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 Kakula-Kansoko Project Mineral Reserve

The Kakula-Kansoko 2020 PFS Project Mineral Reserve includes the ore from both the Kakula Mine and Kansoko Mine at Kamoa. 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 15.3.

Table 15.3 Kamoa-Kakula Project Mineral 2020 Mineral Reserve

Classification	Ore (Mt)	Copper (%)	Copper (Contained Mlb)	Copper (Contained kt)
Proven Mineral Reserve	–	–	–	–
Probable Mineral Reserve	235.0	4.47	23,190	10,519
Mineral Reserve	235.0	4.47	23,190	10,519

1. Effective date of the all Mineral Reserves is 8 September 2020.
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 for Kansoko and $3.10/lb for Kakula.
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.

The Mineral Reserves reported above are not additive to the Mineral Resources.

16 MINING METHODS

16.1 Kakula Geotechnical Investigation and Design

The geotechnical investigation and design work for Kakula was undertaken by SRK Consulting (South Africa) Pty Ltd (SRK) for the Kakula 2020 FS.

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) and the results for the Kakula 2016 PEA were included in this FS investigation.

The data obtained during the geotechnical investigation was used for rock mass classification and geotechnical domaining and this information was used to provide input into the mine design.

Geotechnical designs for room-and-pillar method and the drift-and-fill mining method were carried out by SRK during the FS programme discussed above.

16.1.1 Outline of Work Programme

To complete the study, the following work programme was followed:

- All geotechnical data gathered for Kakula mine was assessed. A detailed programme for further rock strength testing was compiled with a focus on testing of the siltstone, which was underrepresented.
- Multiple site visits have been conducted for the Kakula Project for a review of underground conditions and QA/QC of the geotechnical data.
- Laboratory rock strength testing was conducted by Rocklab (Pretoria) and CUPRUM (Poland). Testing comprised uniaxial compressive strength tests with Young's Modulus and Poisson's Ratio (UCM), uniaxial indirect tensile strength tests (Brazilian method) and triaxial compressive strength (TCS) tests. All test results were analysed and combined with test results obtained during previous studies. An analysis of the point load data gathered on the project site was also carried out.
- Following the pre-feasibility study (PFS), several new holes have been drilled. This includes holes drilled in the area where room and pillar mining has been planned (previously data deficient). From the new holes drilled, there are approximately 150 holes which contain geotechnical data. This new information was analysed and then added to the current geotechnical database which allowed for increased confidence in the current geotechnical model. Based on these results, the location of major structures and the planned mine design, geotechnical domains were identified.
- During the PFS, there were eight drillholes with structural data available for the structural analysis. From the new drilling carried out after the PFS, there are approximately 95 more drillholes with structural logging data. Using this new data and the data from the PFS, a complete geotechnical structural analysis was carried out and structural domains were identified.

3D numerical modelling was undertaken to assess the following:

- Mining sequences, production pillars and standoff zones for the faults.
- The potential for seismic risk that may be associated with the drift and fill mining operations and the geological structures.
- Critical excavations that were not assessed during the execution phase. These planned excavations were also assessed to determine whether they would be adversely affected by stress.
- Modelling to assess the dynamic support requirements that may be required as the mining increases with depth.

The current geotechnical design parameters were assessed, and updated parameters are provided. These include:

- Recommendations for an optimised stope design and sequence based on the numerical analysis.
- Numerical assessment of connection pillar requirements and extraction sequences.
- Backfill strength requirements for single cut and double cut mining operations.
- Stability assessment for the raise bored ventilation shafts.
- Support requirements for excavations not catered for during the execution study.
- Ground support requirements for production drifts, development drives and critical infrastructure.

Overall, 188 geotechnical borehole logs, 42 structural borehole logs and 483 laboratory strength tests were utilised in the feasibility study. Based on the assessments of the data quality it was found that the data available was generally acceptable for use and with minor adjustments was successfully incorporated into the feasibility study as discussed in section 16.1.2.

16.1.2 Geotechnical Database

Geotechnical drilling and logging specific to the Kakula area were conducted by Ivanhoe Mines. SRK completed six site visits to Kamoa during 2016–2019 for the purposes of geotechnical and structural logging quality assurance and quality control (QA/QC) on the Kakula Project.

Findings from the visits were documented in letter reports 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. 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 188 exploration drillholes were provided to SRK in Excel spreadsheets to form the basis of the rock mass classification for the Kakula deposit. 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 14–16 August 2018 to verify the PFS geotechnical data, and between 11–15 November 2019 to verify the feasibility study data. While some opportunities for improvements were identified during the QA/QC reviews, the logging conducted by Kamoa personnel is considered by SRK to be acceptable for a feasibility study.

Laboratory Test Analyses

Laboratory rock strength tests were undertaken for the three major lithologies diamictite (SDT), siltstone (SSL) that form the hanging wall, and the orebody and sandstone (SST) that forms the footwall, as part of the geotechnical investigation. These tests included:

- Uniaxial Compressive Strength (UCS) often with additional measurement of elastic properties (Young's modulus E and Poisson's ratio ν).
- Triaxial Compressive Strength (TCS).
- Uniaxial Indirect Tensile (UTB) Strength (Brazilian method).

This data was used for rock mass classification and input for the numerical modelling.

Two accredited testing facilities were used: Rocklab in Pretoria South Africa, and Cuprum in Poland. 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 and Table 16.2. For UCS testing the tests are considered successful if the failure was through intact rock and not along an existing discontinuity. For Elastic properties such as Poisson's ratio, some laboratory test results are excluded where the Poisson's ratio was out of range. For triaxial compressive strength (TCS), note that the UCS and TCS test results are separated into "all tests" (where samples that have failed along discontinuities are included) and "intact tests" (where the failure was not influenced by discontinuities because the samples were Intact).

Table 16.1 Summary of Elastic Properties Laboratory Testing Data

Material Property	Statistic	Diamictite (SDT)	Siltstone (SSL)	Sandstone (SST)
E (GPa)	Number of tests	109.00	7	49.00
	Intact tests	4.00	0	14.00
	Minimum	60.00	–	68.00
	Mean	61.00	–	74.00
	Maximum	63.00	–	79.00
	Standard deviation	1.00	–	4.00
Poisson's ratio	Number of tests	89.00	2	44.00
	Intact tests	4.00	0	14.00
	Minimum	0.23	–	0.15
	Mean	0.26	–	0.21
	Maximum	0.29	–	0.33
	Standard deviation	0.04	–	0.06

Note:Tests are considered successful if the failure was through intact rock and not along an existing discontinuity. Statistics such as minimum, mean, maximum and standard deviation calculated using intact tests only.

Table 16.2 Summary of Rock Strength Laboratory Testing Data

Material Property	Statistic	Diamictite (SDT)	Siltstone (SSL)	Sandstone (SST)
Density (kg/m^3)	Number of Tests	151	56	57
	Minimum	2.55	2.68	2.25
	Mean	2.82	2.94	2.62
	Maximum	3.04	3.39	2.78
	Standard deviation	0.07	0.13	0.11
UCS (MPa)	Number of Tests	151	56	57
	Intact tests	15	11	20
	Minimum	63	48	90
	Mean	122	140	225
	Maximum	197	228	333
	Standard deviation	41	64	76
TCS10 MPa	Number of Tests	14	0	2
	Intact tests	0	0	0
TCS 20 MPa	Number of Tests	17	2	2
	Intact tests	3	0	0
	Minimum	189	–	–
	Mean	257	–	–
	Maximum	392	–	–
	Standard deviation	117	–	–
TCS 30 MPa	Number of Tests	14	1	0
	Intact tests	1	0	0
	Mean	218	–	–

Note: Statistics such as minimum, mean, maximum and standard deviation calculated using intact tests. Statistics not provided where insufficient data available.

Uniaxial Indirect Tensile (UTB) Strength (Brazilian method) and Base friction angle testing was carried out during the pre-feasibility stage of the study. These results are summarised in Table 16.3.

Table 16.3 Summary of Tensile and Base Friction Angle Laboratory Testing Data

Material Property	Statistic	Diamictite (SDT)	Siltstone (SSL)	Sandstone (SST)
Uniaxial Tensile Brazilian Method (MPa)	Number of tests	86	13	20
	Minimum	2	2	90
	Mean	9	5	225
	Maximum	18	11	333
	Standard deviation	3	3	76
Base friction angle (°)	Number of tests	18	2	13
	Minimum	28	32	28
	Mean	34	35	34
	Maximum	39	37	38
	Standard deviation	3	3	3

In addition to these laboratory tests, point load tests were conducted on site and the results provided to SRK for analysis. The results of the analysis are presented in Table 16.4. The results of the UCS testing and the UCS values determined from the analysis are compared in Table 16.5. The results show that the rock mass comprising the orebody has strong to very strong strength classification in the footwall and sidewall.

Table 16.4 Summary of UCS Results determined from Point Load Strength

	Diamictite (SDT)	Siltstone (SSL)	Sandstone (SST)
Number of tests	566	65	113
Minimum (MPa)	31	47	63
Mean (MPa)	104	96	203
Maximum (MPa)	258	170	411
Standard deviation	35	32	85

Table 16.5 Comparison of UCS from Point Load Tests with UCS from Laboratory Testing

Rock Unit	Mean UCS (MPa)		Strength Classification
	UCS from Point Load Tests	UCS from Laboratory Tests	
Diamictite (SDT)	104	122	Very strong rock
Siltstone (SSL)	96	140	Strong rock / Very strong rock
Sandstone (SST)	203	225	Very strong rock

The information obtained from this assessment was then used as input for rock mass classification and geotechnical domaining as discussed in Section 16.1.3.

16.1.3 Rock Mass Classification and Geotechnical Domaining

To classify the quality of the rock mass in the vicinity of the Kakula orebody, three rock mass classification systems were used: viz. Laubscher's (1990) Mining Rock Mass Rating Classification System, Barton et al.'s (1974) Norwegian Geotechnical Institute's Q-System, and Hoek's (2003) quantification of the Geological Strength Index (GSI) system.

Once rock mass classification results were determined, a weighted averaging method known as compositing was applied to the data, allowing for statistical analysis. This operation was performed using the computer software GEMCOM. Compositing was undertaken for the Kakula orebody (lift 1 and lift 2), the immediate hanging wall (5 m from the top of lift 2) and the immediate footwall (5 m from the bottom of lift 1).

16.1.3.1 Laubscher's (1990) Rock Mass Rating Classification

A summary of the distribution of the Rock Mass Rating (RMR) is presented in Table 16.6. From the RMR results it is observed that most of the rock mass in the Kakula area may be classified as fair rock. This agrees with underground observations made during visits to Kakula mine. The calculations also show that approximately 37% of the rock mass in the orebody is expected to be poor. Note that most of the poor material is likely to be encountered in the western portion of the mining area.

The results of the RMR classification system also show the footwall contains better quality rock compared to the hanging wall and the orebody. This is due to the greater presence of sandstone in the footwall, which is stronger, and was therefore assigned a higher intact rock strength value, compared to the siltstone and the diamictite.

Table 16.6 Summary of RMR Results

Laubscher RMR		Class	5m HW (%)	Orebody (%)	5 m FW (%)
0	20	Very Poor	0	0	0
21	40	Poor	20	37	20
41	60	Fair	69	59	65
61	80	Good	6	3	13
81	100	Very Good	3	1	2

16.1.3.2 The Norwegian Geotechnical Institute's Q-System

The Q-System was utilised to assess the rock mass quality within the vicinity of the Kakula orebody and to determine support recommendations for the mine The Q' contour plots for the orebody, the footwall, and the hanging wall, are presented in Figure 16.1, Figure 16.2, and Figure 16.3. Note that Q' values range between 0.001–1000, whereby a higher Q' value indicates better rock quality.

Figure 16.1 Contour Map of Q' Results for the Orebody (Level 1 and Level 2)

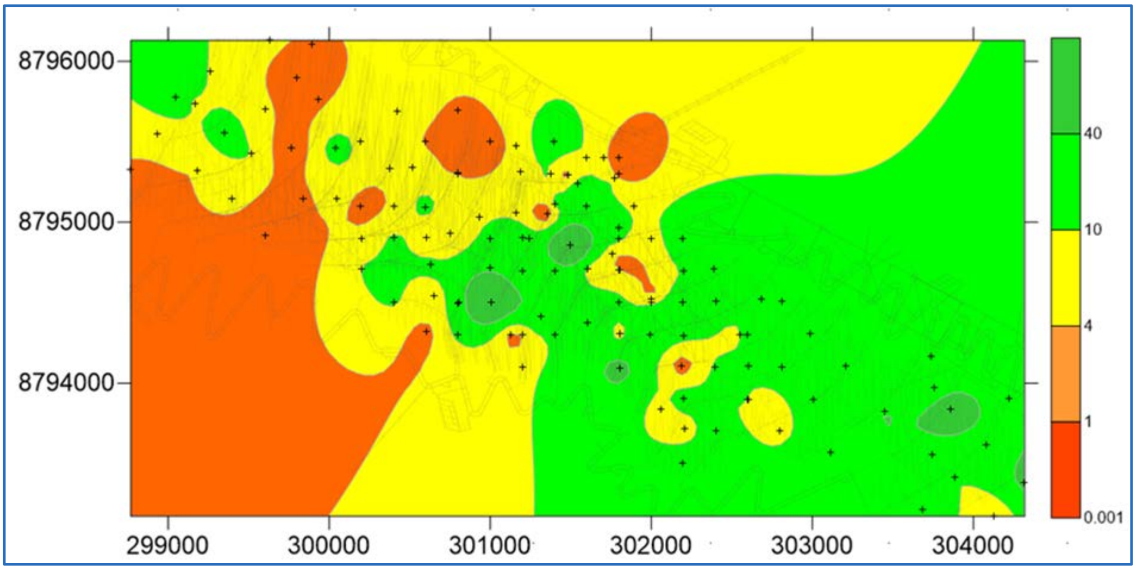


Figure 16.2 Contour Map of Q' Results for the Footwall (5 m)

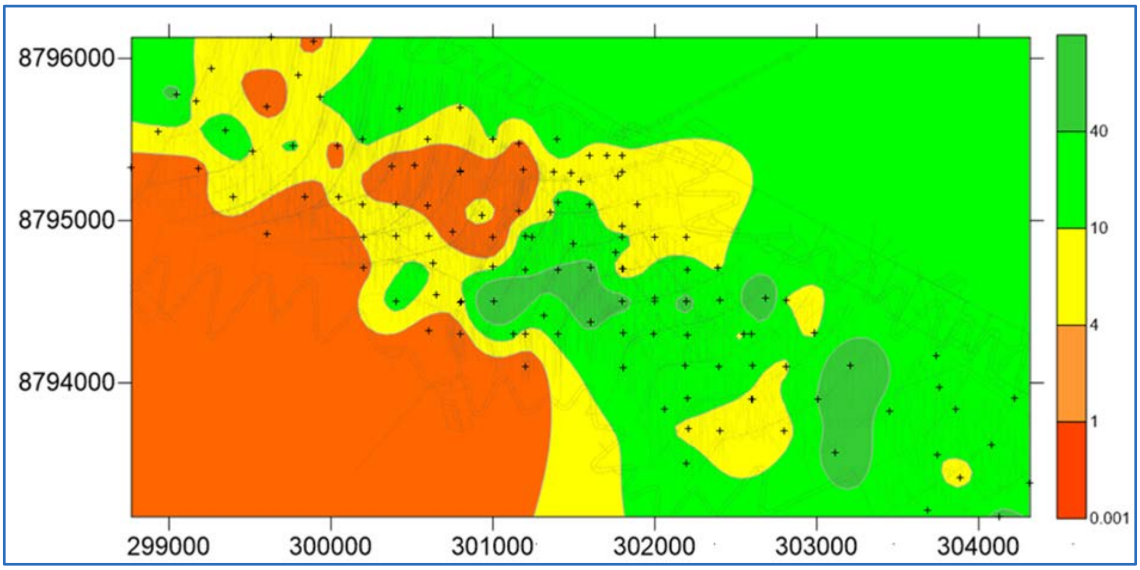
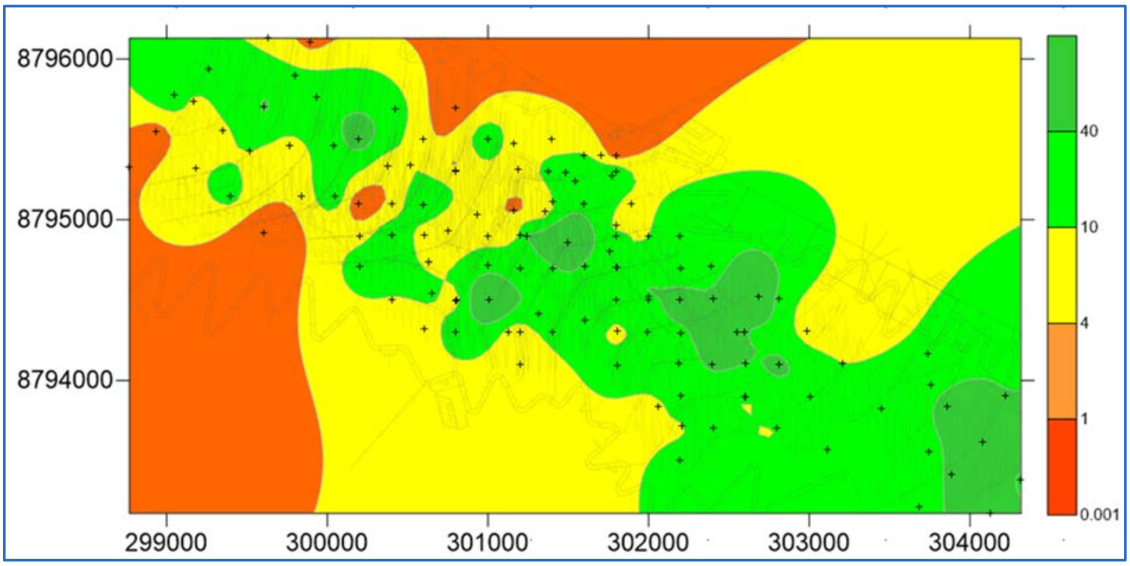


Figure 16.3 Contour Map of Q' Results for the Hanging Wall (5 m)



A summary of the Q' values is presented in Table 16.7. The results from the Q' classification show that most of the rock mass may be classified as fair to good. The calculations also show that 22% of the rock mass that will be encountered in the orebody is expected to be poor. This is lower than the 37% determined for the orebody using Laubscher's rock mass rating system. Furthermore, this is slightly lower than what was determined in the PFS (28%). The differences in the classification systems as well as the additional borehole logs and relogging undertaken are likely the reasons for the differing results.

Table 16.7 Summary of Q' Results

NGI Q' System		Class	5 m HW (%)	Orebody (%)	5 m FW (%)
0.001	1	Very Poor	0	0	0
1	4	Poor	6	22	14
4	10	Fair	33	31	34
10	40	Good	48	42	39
40	100	Very Good	14	5	13

16.1.3.3 Geological Strength Index (GSI)

Application of the GSI system indicates that most of the rock mass quality varies from fair to good. An average GSI of 59 has been calculated for the orebody, which is the same number that was determined in the PFS. As these results have been derived from the GSI equation rather than from face mapping, it is recommended that geotechnical face mapping is carried out as mining progresses, to more accurately determine the GSI of the rock mass. A summary of the GSI values is provided in Table 16.8.

Table 16.8 Summary of GSI Results

	5 m HW	Orebody	5 m FW
Minimum	41	23	22
Mean	69	59	60
Maximum	94	94	94
Standard deviation	11	14	15
Number of boreholes	144	153	133

16.1.4 Geotechnical Domains

The geotechnical characteristics are influenced by the three primary lithological domains, 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 fair, and to a lesser extent, poor ground.
- The applied rock mass classification systems indicate that fair conditions appear to be predominant for the Kakula footprint. The Q-values estimated that over the entire mine area, poor ground conditions account for approximately 20%, fair-ground conditions 30%, good ground conditions 40%, with very good ground conditions the remaining 10%. Underground inspections undertaken during 2018 and 2019 for both the southern and northern areas indicate that fair conditions are predominant in these areas. This also corresponds with results collected from routine underground geotechnical mapping.
- The footwall stratigraphy was characterised 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 the aquifer could influence the stability of excavations in the footwall.

16.1.4.1 Geotechnical Structural Analysis

The Rocscience software (DIPS) was used to determine the joint trends and ultimately determine the structural domains for Kakula mine. Core logging information received by SRK from Ivanhoe was used to carry out the analysis. This involved the creation of stereographic plots per borehole to establish structural trends. Boreholes with similar trends were then combined to create six structural domains. Figure 16.4 shows the structural domains determined from the structural logging.

There are a total of six-joint sets in the Kakula project area. A summary of the joint sets present in each structural domain are presented in Table 16.9. The table shows the orientations of each joint set.

Figure 16.4 Kakula Structural Domains and Position of Geotechnical Boreholes

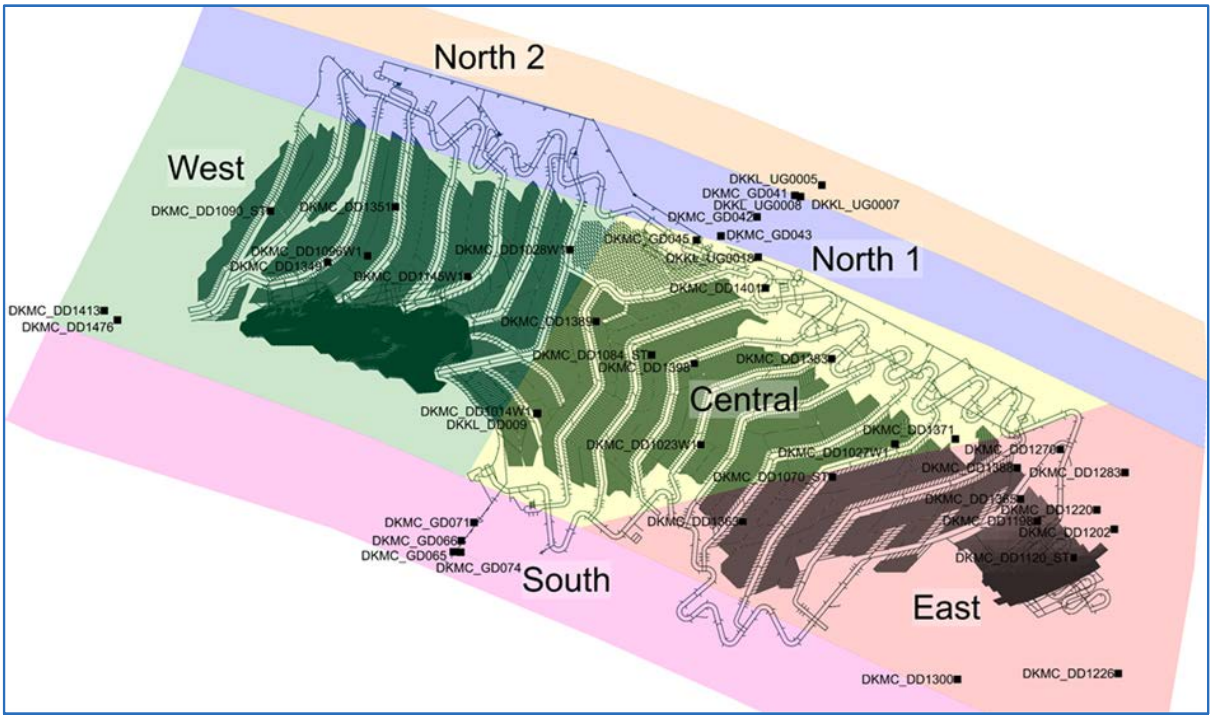


Table 16.9 Summary of Joint Sets in the Kakula Area

Parameter	Range	J1	J2	J3	J4	J5	J6
Dip	Minimum	1	20	38	23	65	75
	Maximum	40	66	70	78	90	90
Dip Direction	Minimum	25	6	221	155	231	180
	Maximum	310	288	284	211	275	198
Number of Joints							
Central		411	–	23	46	–	–
East		81	474	–	–	–	–
West		31	–	83	73	–	–
South		–	–	22	–	–	–
North 1		294	–	–	–	–	–
North 2		22	–	–	–	168	23
Total		839	474	128	119	168	23

16.1.5 Site Visit

The SRK Consulting technical visit for the feasibility study took place in November 2019. Mr Shaun Murphy and Ms Denisha Sewnun represented SRK during this visit which consisted of a quality assessment of the geotechnical logging and several underground visits.

Overall ground conditions appeared to be fair to good and confirm the rock mass classification for Q'. Furthermore, the support installation overall is of a fair quality but in certain areas ravelling is occurring. Additional support should be installed in these areas to limit further damage to the excavations. Support density and compliance should be monitored by rock engineers or strata observers and improved where advised by the Rock Engineering Department.

The results obtained during the rock mass assessment and geotechnical domaining were then used for the geotechnical design.

16.1.6 Geotechnical Analysis and Design

The results obtained from the rock mass assessment was used to carry out numerical assessments to assess the stability of the perimeter drifts and the connection drifts, critical infrastructure and the stability of the protection pillars for the different mining operations.

The mining methods and layouts are described in Section 16.3. The mining methods to be used are room-and-pillar, drift-and-fill, and hanging wall accessed drift-and-fill (HWAD&F). 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.

16.1.6.1 Room-and-Pillar Mining

The findings from the assessment of the room and pillar design are as follows:

- Based on the numerical assessment, the factor of safety for the pillars will be greater than 1.5 m for mining depths between 200–300 m.
- The pillars are diamond shaped with the bull nose likely to scale off during pillar cutting. The bull nose will require Osro straps support to prevent further deterioration.
- The strength factor indicates that the pillars will be stable for mining depths ranging from 200–300 m.
- The assessment shows that stress damage on the access drifts will be minimal as results of room and pillar mining.
- It is recommended that the pillar optimisation is done once underground mapping pillar performance monitoring information becomes available.
- Relatively large intersections are expected based on the design and should be monitored by fall of ground (FOG) lights and borehole camera inspections.

16.1.6.2 Perimeter Drifts and Infrastructure

The numerical assessment indicates that there are no fatal flaws in the placement of the perimeter drifts and the ancillary excavations and any stress interaction that occurs is not significant. It can also be concluded that depth of failure obtained for the excavations will be adequately catered for by the recommended support system.

The big crusher chamber and the transfer bin indicate that stress relaxation on these excavations will be insignificant. For specific large chambers, separate design and considerations will have to take place in order to check the 3D dimensions and analyse location, ground conditions, and the results obtained from the numerical modelling.

To ensure the long-term stability of hanging wall accessed drift-and-fill (HWAD&F) access (both east and west) the infrastructure additional support consisting of yielding tendons and cable anchors will be required.

From a 500 m depth the tertiary drifts-and-pillar are close to failure prior to the extraction of this drift. It must also be noted that the ore pillar prior to tertiary extraction will have a potential to fail throughout statically due to stress fracturing or dynamically by strain bursting. This situation is aggravated as the depth of extraction increases. Tight fill will assist in containing this. As deformation increases with depth some form of yielding tendons will be required to contain the increase of deformation, specifically when extracting the secondary drift. No yielding tendons will be required during the extraction of the tertiary drift. Further use of rib pillars should be considered to contain the effects of the stress levels as that will increase with the depth of mining.

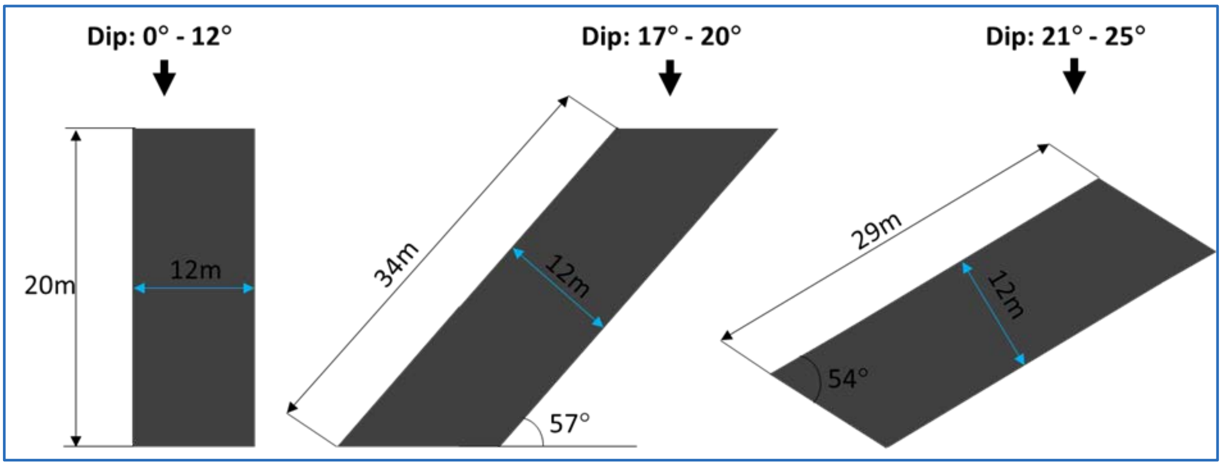
Table 16.10 summarises the connection drift pillar length for changing orebody dips. The orebody dip ranging from 0–12° is considered flat and the length of the pillar will be perpendicular to the connection drift. For ore body dips greater than 12° the length of the pillar will be adjusted on dip to maintain the plan length between the connection drifts and the production drifts. The objective of this is to increase the length of the pillar to reduce the reliance on the apex. Trusses should be installed to contain any potential failure occurring in the apex. Figure 16.5 is an example showing the pillar configuration for flat dipping 17–20° and 21–25° dips. The connection drift will be oriented on strike for dip angles from 0–20° and will be oriented on apparent dip not greater than 12° for dips greater than 20°.

Table 16.10 Connection Drifts Production Pillars

Depth (m)	Plan Pillar Length for dip Ranges (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 on apparent dip.

Figure 16.5 Example of Production Pillar Configuration for Varying Orebody Dip



16.1.6.3 Production Pillar Extraction

The production pillar extraction strategy uses a retreat methodology which requires strict adherence to this methodology.

Support for the stability of both the hanging wall and the sidewalls is of paramount importance in this scenario and must be strictly adhered to.

Reduction in the size of each pillar and the associated increase in pillar stress during the extraction process will be contained to a great extent by the strict adherence to the extraction sequence backfill confinement and the support installed. However, as the pillar size decreases, crushing of smaller pillars, or dynamic failure of the pillars may occur. To assist in the extraction process, the following is recommended:

- Yielding type of support installed during pillar extraction phase, to cater for deformation and dynamic load.
- Use the higher-binder content in order to increase backfill strength for stopes mined during ore recovery operations.
- Use remote controlled mobile equipment for the process.
- Monitoring of the deformation that occurs on an ongoing programme.

16.1.6.4 Double Cut Mining

It must be noted that any pillars left in-situ below the second lift will result in the second lift mining being compromised by the high stress field associated with the pillar left in-situ on the first lift.

If tight filling is not accomplished on the first lift horizon, significant bed separation and ravelling will be encountered when mining the second lift.

The accessways to the second lift must not be developed through the production pillar on the first lift and no further mining of the production pillars left on the first lift for the protection of the second lift accessway. These pillars can be extricated after the second lift mining has been completed.

16.1.7 Backfill

The calculated required backfill strength is 130 kPa to achieve a free-standing height of 7.0 m. The secondary and tertiary drifts backfill sidewalls will also be required to be free standing.

Backfill strength at Kakula has been designed to have a strength of 200 kPa UCS for single-cut option, 500 kPa for bottom-cut, and 200 kPa for upper-cut where the orebody thickness is greater 6 m and double lift mining is required. The geotechnical criteria are comfortably catered for with these criteria.

Tight filling (high as possibly can practically be achieved) should be implemented and monitored. Up-dip or down-dip extraction and backfilling scenarios as well as sequential backfilling in shorter sections could help to improve the tight filling of the excavations.

QA/QC control on the backfill properties should be implemented on-site. Mine site strength testing, particle size distribution and the overall rheology of the mix needs to be monitored. Controlling water cement ratios and producing a backfill mix with consistent slump is important to minimise the increase of friction inside the pipes. The paste fill plant must be equipped with a testing facility to provide all the monitoring requirements discussed above.

16.1.8 Subsidence

Due to the contiguous placement of backfill in adjacent drifts will be confined and will therefore increase resistance as the deformation increases. Stope hanging wall deformation is therefore expected to be purely elastic as determined in the section above. Surface deformation, 200 m above the stope, will therefore be negligible. Considering the low to negligible subsidence that will occur, the seismic response is expected to be minimal.

16.1.9 Monitoring of Potential Seismic Activity

The major structures, namely the west fault structures contain soft infilling, and this coupled with tight filling during the drift and fill mining operations, it is unlikely that seismic activity will occur on these structures. The planned bracket pillars that will be left along the large structures will also assist in containing seismic activity on these structures. However, an induced localised seismic response associated with strain bursting and or pillar bursting may occur. This will be contained as a result of tight filling and correct sequencing during the cut and fill mining operation. As the depth increases below 500 m the potential of this localised seismic activity increases and it will be prudent to install a seismic system.

16.1.10 Ground Support

Ground support requirements have been determined and the recommended support types are shown in Table 16.11 to Table 16.16.

Table 16.11 Access, Perimeter Drifts, Conveyor Drive, and Workshops

Conditions	Support
Perimeter Drifts (Diamictite)	
Good conditions	2.4 m long resin bar bolt in a 1.5 m x 1.5 m pattern on the hanging wall and the sidewalls + 25 mm of fibrecrete.
Fair conditions	2.4 m long resin bar bolt in a 1.5 m x 1.0 m pattern on the hanging wall and the sidewalls + 50 mm of fibrecrete.
Poor and very poor	3.0 m long resin bar bolt in a 1.0 m x 1.0 m pattern on the hanging wall and the sidewalls + 50 mm of fibrecrete.
Intersections	6.5 m long cable anchors (pattern offset from primary support).
Conveyor Drive (Upper Diamictite)	
Good conditions	3.0 m long resin bar bolt in a 1.5 m x 1.5 m pattern on the hanging wall and the sidewalls + 25 mm of fibrecrete.
Fair conditions	3.0 m long resin bar bolt in a 1.5 m x 1.0 m pattern on the hanging wall and the sidewalls + 50 mm of fibrecrete.
Poor and very poor	3.0 m long full column resin bar bolt in a 1.0 m x 1.0 m pattern on the hanging wall and the sidewalls + 50 mm of fibrecrete.
Intersections	6.5 m long cable anchors (pattern offset from primary support).
Underground Workshops, Diamictite	
Good conditions	2.4 m long resin bar bolt in a 1.5 m x 1.5 m pattern on the hanging wall and the sidewalls + 25 mm of fibrecrete.
Fair conditions	3.0 m long resin bar bolt in a 1.5 m x 1.0 m pattern on the hanging wall and the sidewalls + 50 mm of fibrecrete.
Poor and very poor	3.0 m long resin bar bolt in a 1.0 m x 1.0 m pattern on the hanging wall and the sidewalls + 50 mm of fibrecrete.
Intersections	6.5 m long cable anchors (pattern offset from primary support) and for the excavations where width is ≥8.5 m.

Table 16.12 Connection Drifts

Conditions	Support
Twin Connection Drifts (Depth <500 m)	
Good conditions	3.0 m long resin bar bolt in a 1.5 m x 1.5 m pattern with mesh on the hanging wall and the sidewalls.
Fair conditions	3.0 m long resin bar bolt in a 1.5 m x 1.0 m pattern with mesh on the hanging wall and the sidewalls.
Poor and very poor	3.0 m long resin bar bolt in a 1.0 m x 1.0 m pattern with mesh on the hanging wall and the sidewalls.
Intersections	6.5 m long cable anchors (pattern offset from primary support).
Twin Connection Drifts (Depth >500 m)	
Good conditions	3.0 m long yielding bar bolt in a 1.5 m x 1.5 m pattern with mesh on the hanging wall and the sidewalls.
Fair conditions	3.0 m long yielding bar bolt in a 1.5 m x 1.0 m pattern with mesh on the hanging wall and the sidewalls.
Poor and very poor	3.0 m long yielding bar bolt in a 1.0 m x 1.0 m pattern with mesh on the hanging wall and the sidewalls.
Intersections	6.5 m long cable anchors (pattern offset from primary support).

Table 16.13 Room-and-pillar Mining

Conditions	Support
Room-and-pillar Mining (6 m W x 4.5 m H)	
Good conditions	2.4 m long resin bar bolt in a 1.5 m x 1.5 m pattern on the hanging wall and the sidewalls.
Fair conditions	3.0 m long resin bar bolt in a 1.5 m x 1.5 m pattern on the hanging wall and the sidewalls.
Poor and very poor	3.0 m long resin bar bolt in a 1.5 m x 1.0 m pattern on the hanging wall with OSRO straps and mesh installed on the pillars' corners.
Intersections	6.5 m long cable anchors (pattern offset from primary support) – only for poor and very poor ground conditions, for good and fair conditions monitoring system with warning lights, and extensometers will be installed on each intersection.

Table 16.14 Drift-and-fill Mining

Conditions	Support	
Drift-and-fill Mining		
Good conditions	Block access	2.4 m long resin bar bolt in a 1.5 m x 1.5 m pattern on the hanging wall and the sidewalls.
	Central intersection <8.5 m wide	2.4 m long resin bar bolt in a 1.0 m x 1.0 m pattern on the hanging wall and the sidewalls.
	Primary	2.4 m long split-set bolt in a 1.5 m x 1.5 m pattern on the hanging wall and the sidewalls.
	Secondary	2.4 m long split-set bolt in a 1.5 m x 1.5 m pattern on the hanging wall and the sidewalls + Backfill support: 1.8 m long split-sets in a 2.0 m x 2.0 m pattern with mesh.
	Tertiary	
Fair conditions	Block access	3.0 m long resin bar bolt in a 1.5 m x 1.5 m pattern on the hanging wall and the sidewalls.
	Central intersection <8.5 m wide	3.0 m long resin bar bolt in a 1.0 m x 1.0 m pattern on the hanging wall and the sidewalls.
	Primary	3.0 m long split-set bolt in a 1.5 m x 1.5 m pattern on the hanging wall and the sidewalls.
	Secondary	3.0 m long split-set bolt in a 1.5 m x 1.5 m pattern on the hanging wall and the sidewalls + Backfill support: 1.8 m long split-sets in a 1.5 m x 1.5 m pattern with mesh.
	Tertiary	
Poor and very poor	Block access	3.0 m long full column resin bar bolt in a 1.0 m x 1.0 m pattern with mesh on the hanging wall and sidewalls.
	Central intersection <8.5 m wide	3.0 m long full column resin bar bolt in a 1.0 m x 1.0 m pattern with mesh on the hanging wall and sidewalls.
	Primary	3.0 m long split-set bolt in a 1.0 m x 1.0 m pattern with mesh on the hanging wall and the sidewalls.
	Secondary	3.0 m long split-set bolt in a 1.0 m x 1.0 m pattern with mesh on the hanging wall and the sidewalls + Backfill support: 1.8 m long split-sets in a 1.5 m x 1.5 m pattern with mesh.
	Tertiary	

Table 16.15 Hanging Wall Access Drift-and-fill Mining

Conditions	Support	
Hanging Wall Access Drift-and-fill (HWAD&F)		
Good conditions	Twin-decline drifts or single-decline	2.4 m long yielding bolt in a 1.5 m x 1.5 m pattern on the hanging wall and the sidewalls + 50 mm of fibrecrete.
	Level developments	2.4 m long yielding bar bolt in a 1.5 m x 1.5 m pattern on the hanging wall and the sidewalls + mesh.
	Attack ramps	2.4 m long yielding bar bolt in a 1.5 m x 1.5 m pattern on the hanging wall and the sidewalls.
	Intersections	Primary support + cable bolting (pattern 3.0 m x 2.5 m).
	Production drifts	2.4 m long split-set bolt in a 1.5 m x 1.5 m pattern on the hanging wall and the sidewalls+ Backfill support (for west area only): 1.8 m long split-sets in a 2.0 m x 2.0 m pattern with mesh.
Fair conditions	Twin-decline drifts or single-decline	2.4 m long yielding bolt in a 1.5 m x 1.0 m pattern on the hanging wall and the sidewalls + 50 mm of fibrecrete.
	Level developments	2.4 m long yielding bar bolt in a 1.5 m x 1.0 m pattern on the hanging wall and the sidewalls + mesh.
	Attack ramps	2.4 m long yielding bar bolt in a 1.5 m x 1.0 m pattern on the hanging wall and the sidewalls.
	Intersections	Primary support + cable bolting (pattern 2.0 m x 2.5 m).
	Production drifts	2.4 m long split-set bolt in a 1.5 m x 1.5 m pattern on the hanging wall and the sidewalls + Backfill support (for west area only): 1.8 m long split-sets in a 2.0 m x 2.0 m pattern with mesh.
Poor and very poor	Twin-decline drifts or single-decline	2.4 m long yielding bolt in a 1.0 m x 1.0 m pattern on the hanging wall and the sidewalls + 50 mm of fibrecrete.
	Level developments	2.4 m long yielding bar bolt in a 1.0 m x 1.0 m pattern on the hanging wall and the sidewalls + mesh.
	Attack ramps	2.4 m long yielding bar bolt in a 1.0 m x 1.0 m pattern on the hanging wall and the sidewalls.
	Intersections	Primary support + cable bolting (pattern 2.0 m x 2.0 m).
	Production drifts	2.4 m long split-set bolt in a 1.5 m x 1.5 m pattern on the hanging wall and the sidewalls + Backfill support (for west area only): 1.8 m long split-sets in a 2.0 m x 2.0 m pattern with mesh.
	Ore passes	Geotechnical drilling required in order to determine ground conditions, 75 mm shotcreting with high strength wearing resistant mix (HSWR).

Table 16.16 Technical Specifications for Support Elements

Support Type	Technical Specification
Steel rebar for resin bolts	Minimum yield strength 500 MPa black steel, minimum 22 mm diameter. 250 mm square, domed bearing plates. Bolt annulus maximum 5 mm.
Yielding bars	Dynamic support system allows for up to 200–300 mm bolt elongation
Cable anchor	Minimum 18 mm diameter steel cable, minimum 350 kN ultimate load, 300 mm domed bearing plates. Pretensioning required.
Split-set bolts	For backfilled sidewalls: SS39 size, installed in 36 mm drill hole diameter. For hanging wall: SS42 size installed in 39 mm diameter drill hole.
Mesh	Black welded steel mesh, minimum 5 mm gauge, maximum 100 mm aperture.
Capsule resins	Two component urethane silicate resin capsules. Fast set <30 seconds and slow 5–10 minutes setting times.
Cement grout	Minimum 40 MPa Ordinary Portland Cement, water cement ratio 0.35:0.40.
Fibrecrete	Polyurethane fibre-reinforced shotcrete, minimum 30 MPa strength (28 days), fibres (4–6 kg/m^3).
OSRO straps	Steel straps, 250 mm wide, configuration with 4 rods, rod thickness 8–10 mm.

16.1.11 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 FS requirements and no fatal flaws were identified during the FS.

The geotechnical risks for this project were identified and are summarised below:

- The stress environment is unknown at this stage and the numerical modelling was done using a k ratio of one. If the horizontal stress is significantly higher than anticipated an increase in the depth of failure in the hanging wall of the long-term excavations could occur. It must, however, be stated that the initial technical visit shows no evidence of this occurring during observation. Consideration should be given to include in-situ stress tests during the execution phase.
- 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 was assumed during this study, where limited overbreak occurs. Poor blasting will result in smaller and taller than designed pillars in both the room and pillar and the cut and fill mining operations, negatively impacting on the pillar and span stability. Failure to achieve good quality blasting will significantly affect pillar performance.
- 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 hanging wall.

16.2 Kansoko Geotechnical Investigation and Design

The geotechnical investigation for Kamoa is based on geotechnical drilling and logging conducted by Ivanhoe Mines over the Kamoa project area, which was reviewed and interpreted by SRK. The work is essentially unchanged from the previous studies on Kamoa (Kamoa 2013 PEA, Kamoa 2016 PFS) undertaken in order to provide geotechnical designs for the room-and-pillar method incorporated in the Kamoa 2017 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.

16.2.1 Geotechnical Database

Geotechnical drilling and logging were conducted by Ivanhoe. 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 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 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.

16.2.2 Geotechnical and Structural Models

16.2.2.1 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.

16.2.2.2 Joint Patterns

There are three 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.
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 hanging wall 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

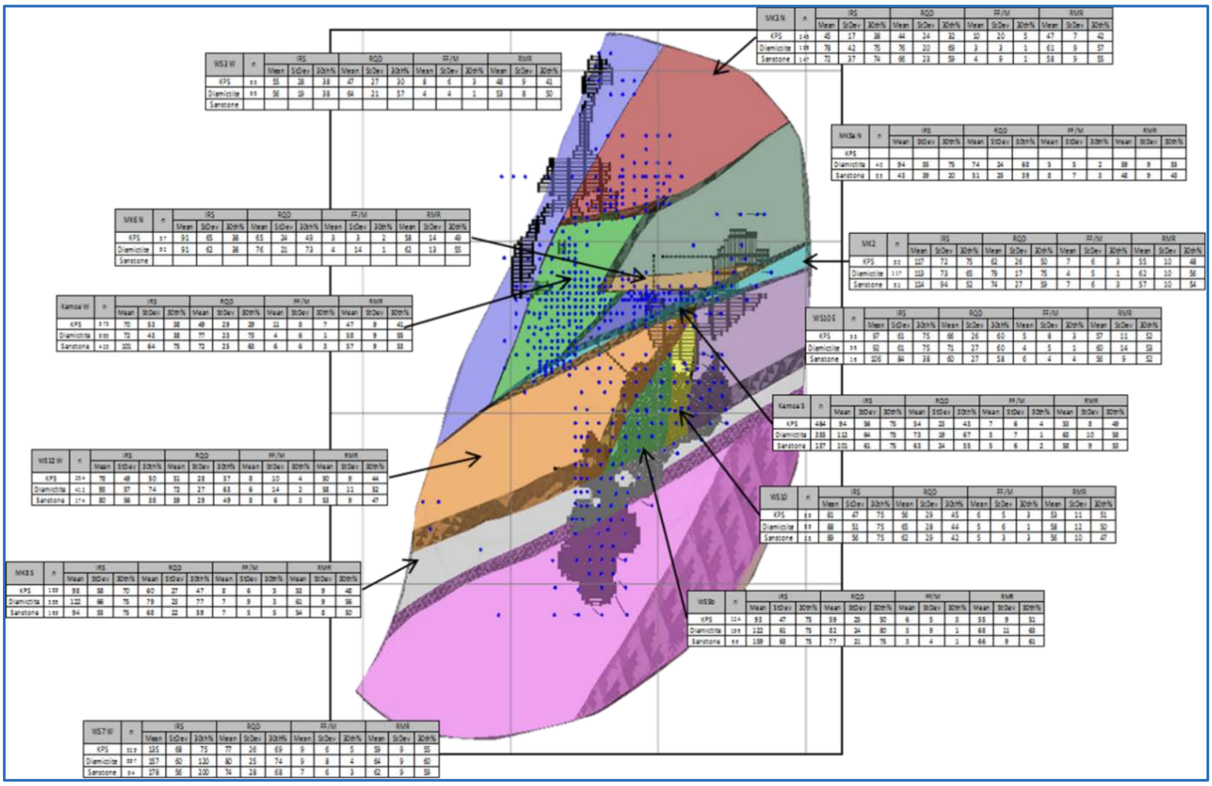


Figure 16.7 Overview of Joint Pattern Variations Across the Kamoa Project, Mapped Outline in Background

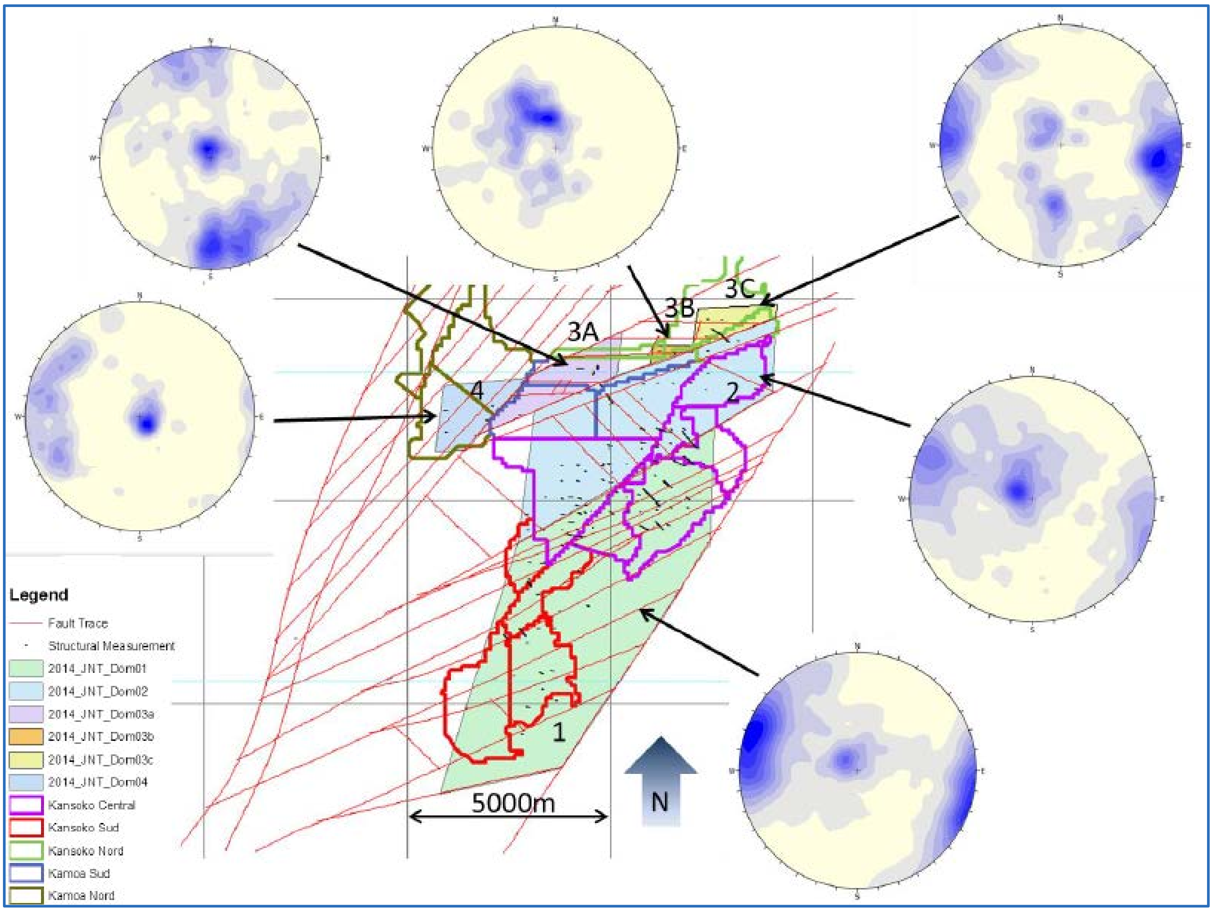


Figure by SRK, 2017.

16.2.3 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 tends to weather rapidly and the distance between the roof and this layer varies considerably and, in some areas, it forms the hanging wall 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.

16.2.4 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 16.17. 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)

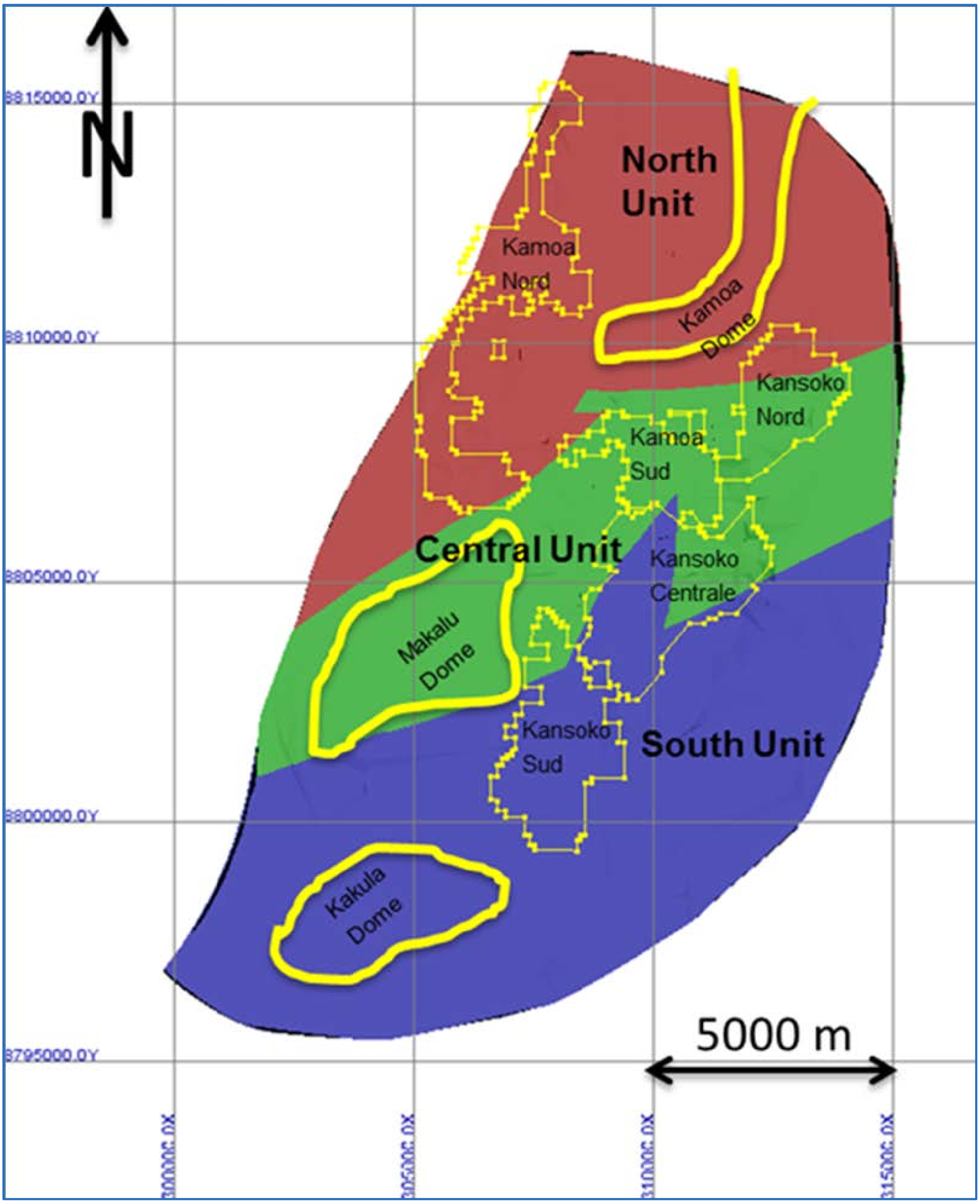


Figure by SRK, 2017.

Table 16.17 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

16.2.5 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 16.18 considers the field estimated IRS (as logged by African Mining Consultants), field point load testing, and laboratory unconfined compressive strength testing. Table 16.18 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 16.18 Summary of Intact Rock Strength Estimates per Geotechnical Domain

Domain	Stratigraphy	Logged IRS (MPa)	No. of UCS Tests	Intact UCS (MPa)	Point Load Test (MPa)		Engineered IRS (MPa)
					Axial	Diametral	
Weathered	KPS	45 (32)	1	123 (xx)	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

Note: Values shown in parenthesis are standard deviation.

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 that are using the controlled convergence room-and-pillar mining method.

16.2.6 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) hanging wall closure, and rests on the principle of crushed pillars providing a residual support capacity directly after being cut at the advancing face.

16.2.7 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 has 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 and 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 hanging wall, 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 W and 500 m.

Overall, the controlled convergence room-and-pillar methodology appears to be suitable for the Kansoko mining area, however, a focussed assessment of the hanging wall 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.

16.2.7.1 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 the 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 16.19.

Table 16.19 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

16.2.7.2 Pillar Design (Controlled Convergence Room-and-Pillar)

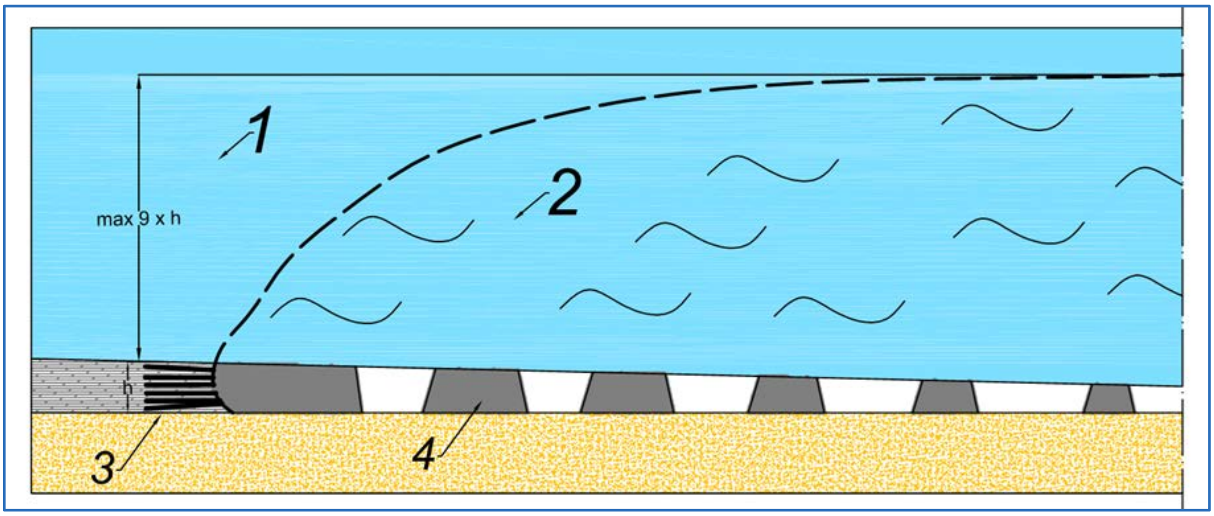
Pillar strength for the controlled convergence room-and-pillar mining method is premised on a percentage of the UCS for the post-failure strength estimate. The results of 14% and 16.5% of the UCS are reasonable quantities (~20 MPa) of post failure strength. This is the strength assigned to the pillars (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 16.20.

The anticipated mode of failure and deformation of the hanging wall 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 considers 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 provisions made in the form of recommendations that include hydraulic props, wooden cribs and cable bolts must be used in areas where complex geological or difficult mining conditions exist.

Table 16.20 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.

16.2.7.3 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 13–16°, 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 and 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 considered 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°

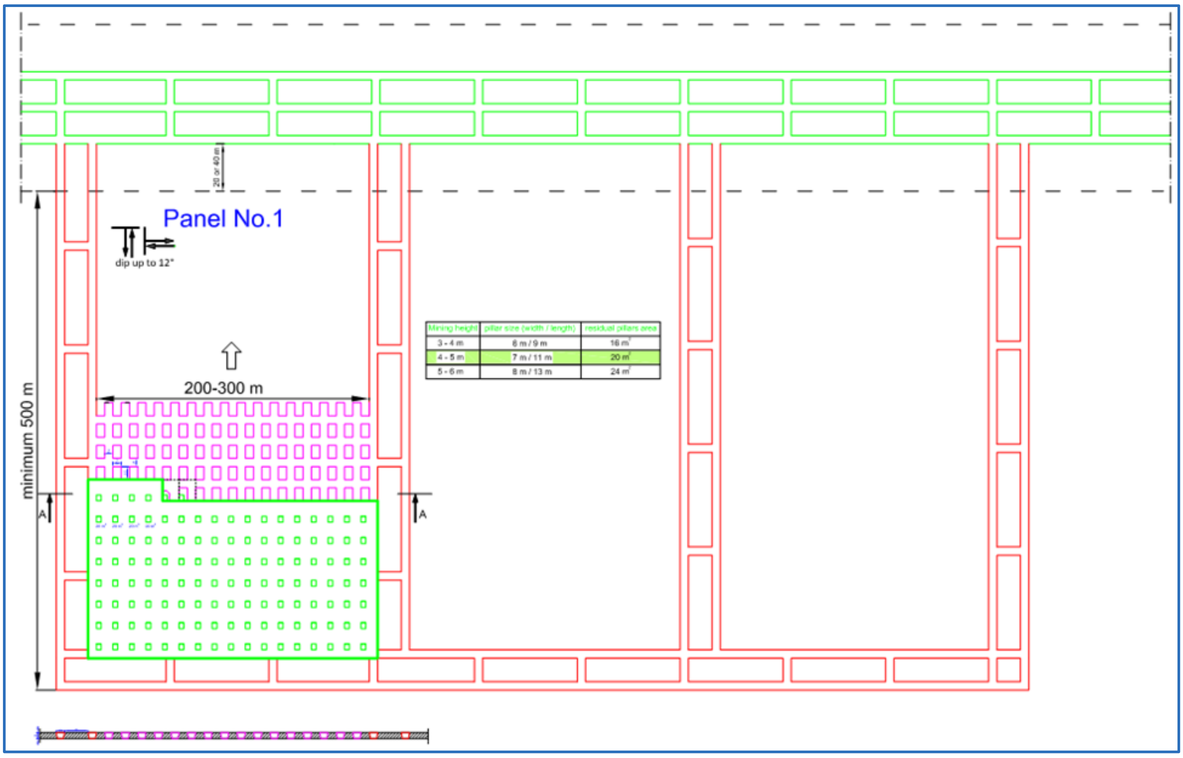


Figure by KGHM Cuprum, 2017.

Figure 16.11 Controlled Convergence Room and Pillar Mining Method and Pillar Geometry for a Deposit with Dip Angle of 13–16°

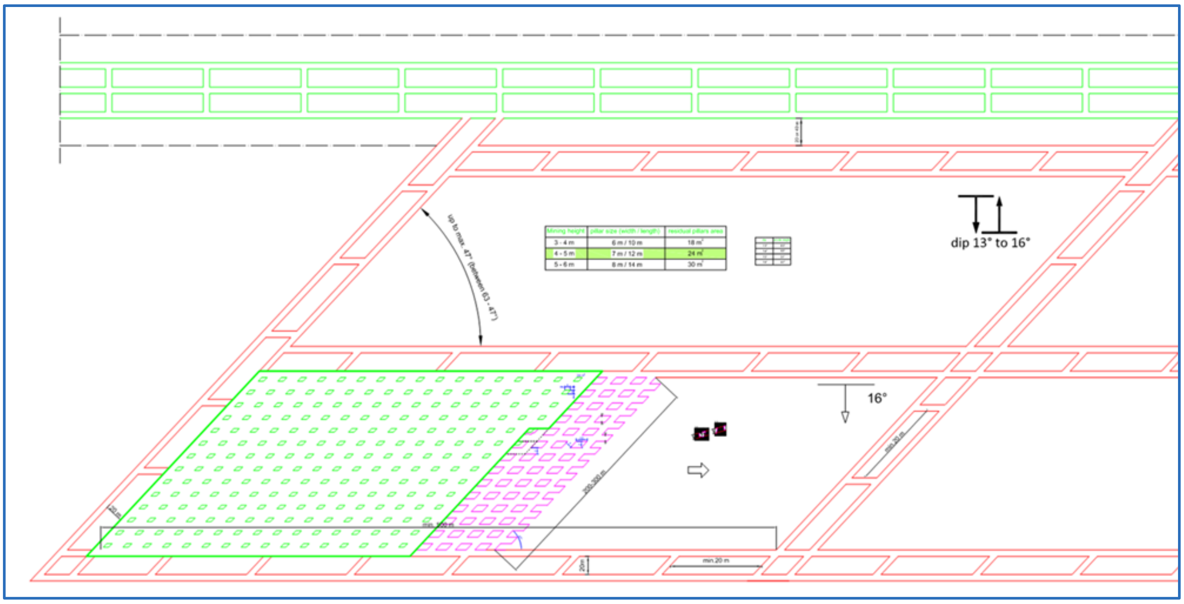


Figure by KGHM Cuprum, 2017.

16.2.8 Numerical Analysis

The two-dimensional software programme Phase 2 (version 8) was used by Cuprum to conduct numerical modelling to determine the minimum width of the protection and 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 and 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 hanging wall 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 hanging wall, 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.

16.2.9 Support Design

Support design uses the principle of the height of a de-stressed “tensile” zone within the hanging wall 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 hanging wall closure will increase, it is important to understand if the hanging wall 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 hanging wall 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 (FoG) may occur in the back area. Such FoG 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.

16.2.10 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.

16.2.11 Water Management

This is discussed in the Cuprum (2016) report however the management of water is not specifically addressed. The goafing of the hanging wall 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 12–30% in different modelled scenarios. It is important that this is taken into consideration in the design.

16.2.12 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 hanging wall, 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 one. However, if the horizontal stress is significantly high this will could result in an increase in the depth of failure in the hanging wall 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 hanging wall 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 hanging wall 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 hanging wall.
- It is important to retreat towards a stable access during pillar extraction to ensure safety of personnel.

16.2.13 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 hanging wall 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 hanging wall 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, 92 p) 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 and 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 hanging wall, 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 one. However, if the horizontal stress is significantly high this could result in an increase in the depth of failure in the hanging wall 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.

16.3 Kakula Underground Mining

16.3.1 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 maximising ore recoveries and production grades, while reducing operating costs. The mine schedule focuses on optimising mining block sequencing, maximising 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 16.21.

Table 16.21 Kakula Probable Mineral Reserves by Mining Method

Production by Mining Method	Ore (Mt)	NSR20 (S/t)	TCu (%)	Fe (%)	As (%)	S (%)
Ore Development	11.0	243	5.22	4.91	0.00	1.43
Drift-and-Fill	80.4	240	5.16	4.77	0.00	1.44
Pillar Extraction	17.2	263	5.64	4.72	0.00	1.55
Room-and-Pillar	1.4	162	3.52	4.79	0.00	0.85
Total Ore*	110.0	243	5.22	4.77	0.00	1.45

*May not sum to total due to rounding.

16.3.2 Mine Design Parameters

16.3.2.1 Ore and Waste Properties

The Kakula deposit is a large stratiform copper deposit, typical of sediment-hosted deposits. The deposit is tabular, with dips varying from 0–58° and thicknesses varying from 3 m to 18 m averaging 8.66 m at a $100 NSR (net smelter return) cut-off. The ore zone density has been defined as using a greater than 2.4% TCu (total copper grade) cut-off. The swell factor for development is 50%.

Table 16.22 details the bulk density parameters of the ore and surrounding waste rock of the Kakula deposit.

Table 16.22 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
Hanging wall	2.39	3.18	2.80
Footwall	2.21	3.04	2.67

16.3.3 Mine Planning

16.3.3.1 Lateral Development

The north decline has a maximum gradient of ±10°. The decline is 7.0 m W x 6.0 m H with a flat back for the conveyor drift, and 5.5 m W x 6.0 m H with 1.5 m arch corners for the service drift.

The south decline is 7.5 m W x 6.0 m H with 1.5 m arch corners and have a maximum gradient of ±9.5°. All main decline drifts have re-mucks located every 150 m, and crosscuts located every 300 m.

All lateral development (such as infrastructure access) is 5.5 m W x 6 m H, with 1.5 m arch corners unless otherwise specified.

Conveyor drifts have a maximum gradient of ±10°. They are 7 m W x 6 m H (1.5 m arch corners), with re-mucks located every 300 m.

All perimeter drifts are 5.5 m W x 6 m H (1.5 m arch corners) with a maximum gradient of ±8.5°. Perimeter service drift development consists of two parallel drifts with re-mucks located every 150 m and cross-cuts every 300 m.

Twinned connection drifts are driven across the targeted ore body from the perimeter declines. Connection drifts are 6 m W x 6 m H with a maximum gradient of ±8.5°. Connection drifts have a flat back.

16.3.3.2 Vertical Development

Vertical development consists of ventilation raises, bins, and boreholes. All ventilation raises are excavated with a raisebore drill. Ventilation shaft 1 (VS1) is 5.5 m in diameter, the ventilation shaft at the workshop (VS-NW1), and the ventilation shaft at the crusher (VS-NE1) are 5.5 m in diameter. All other raises are 6.0 m in diameter. Twin access drifts to the ventilation shafts are 6.0 m W x 6.0 m H.

Ore bins are excavated as drop raises using long-hole drills. Boreholes for paste fill and other services to the underground are drilled from surface using surface drills. These boreholes are cased as required for their purpose.

16.3.3.3 Room-and-Pillar

Room-and-pillar mining issued in the initial mining area of the Kakula deposit. The mining panel width and length limits are determined by production requirements. The height of the panel is from 4.5–6.0 m; the height varies depending on the dip of the ore deposit and mining thickness. Main drifts run parallel to the strike of the panel for dips less than 20°. Cross-cut drifts run at an acute angle to the main drifts to ensure the grade of the main drifts remain less than 10°. For dips less than 12°, cross-cuts are oriented perpendicular to the main drifts. For dips greater than 12° (up to 35°), pillars are diamond-shaped. Cross-section shapes will be rectangular (vertical on sidewalls) and their width are 6 m in plan view.

16.3.3.4 Drift-and-Fill

The majority of mining at Kakula is drift-and-fill. A typical mining block with production drifts perpendicular to the connection drift is 216 m wide as shown in Figure 16.12. There are no barrier pillars required between blocks. The orebody width (north to south) determines the number of mining block. The minimum mining height is 3.0 m measured on the central axis and the maximum height is 6.0 m measured on the central axis.

Each mining block consist of three mining units. Each mining unit comprises four primary headings, four secondary headings, and four tertiary headings.

The production drift cross-section shape has vertical walls up to a maximum height of 6.0 m. Where the ore does not have a second lift, the back of the drift is shantied to the dip of the ore.

For dips less than 12°, block access drifts are oriented perpendicular from the connection drifts. For areas dipping greater than 12°, block access drifts are angled such that development inclination grade does not exceed its maximum limit (10°).

The block access drift cross-section shape has vertical walls, and the hanging wall drift is 6.0 m W x 6.0 m H. The first round will be at a ±3.0° gradient.

The length and width of a mining block is dependent on the dip and depth to maintain an 85% extraction ratio of drift and fill tonnes to production pillars tonnes. Mining block sizes are detailed in Table 16.23.

Figure 16.12 Drift-and-Fill Design (Plan View)

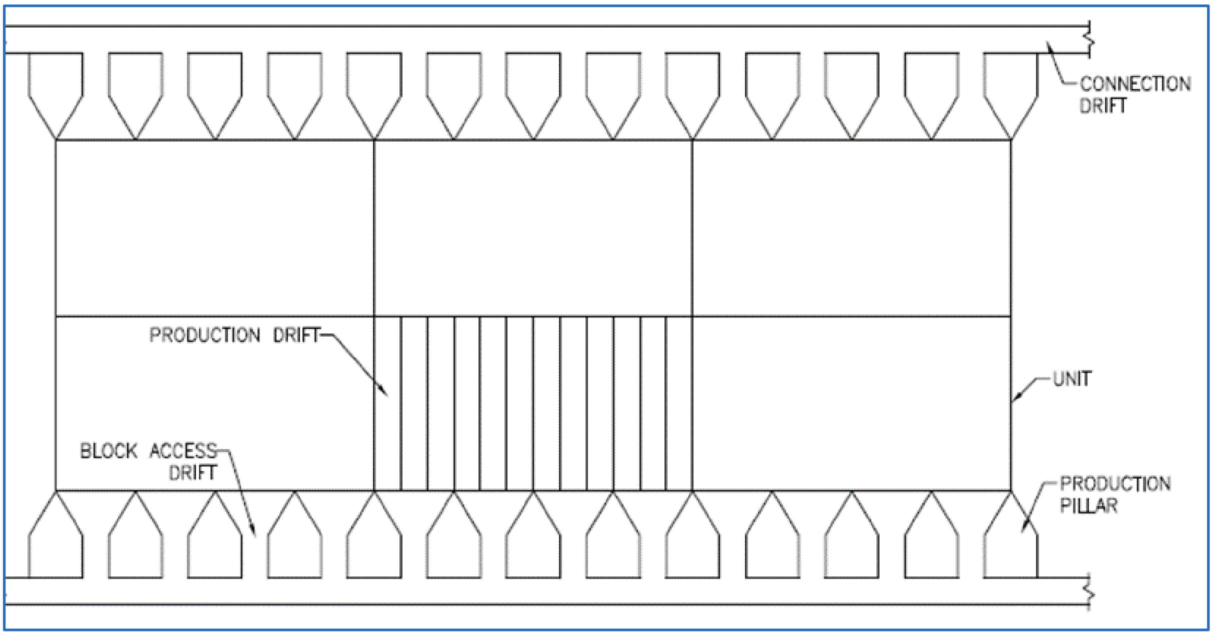


Figure by Stantec, 2020.

Table 16.23 Mining Block Sizes

	Block Sizes					
	85% Extraction Ratio					
Dip	**0°–12°**	**>12°–16°**	**>16°–20°**	**>20°–25°**	**>25°–30°**	**>30°–35°**
Apparent Angle (°)	90	57	40	52	46	36
Mining Block Width (m)	216.0	257.6	336.0	336.0	336.0	490.0
Depth (m)	**Drift Length (m)**					
0–400	99.2	100.9	103.0	101.5	102.2	103.6
400–600	121.4	123.4	125.9	124.1	124.9	126.5
600–800	143.7	145.9	148.7	146.6	147.6	149.4
800–1,000	165.9	168.4	171.5	169.2	170.3	172.3

16.3.3.5 Production and Connection Pillars

Drift-and-fill minimum protection pillar sizes for the block access production pillars and the pillars in between the twinned connection drifts are presented in Table 16.24.

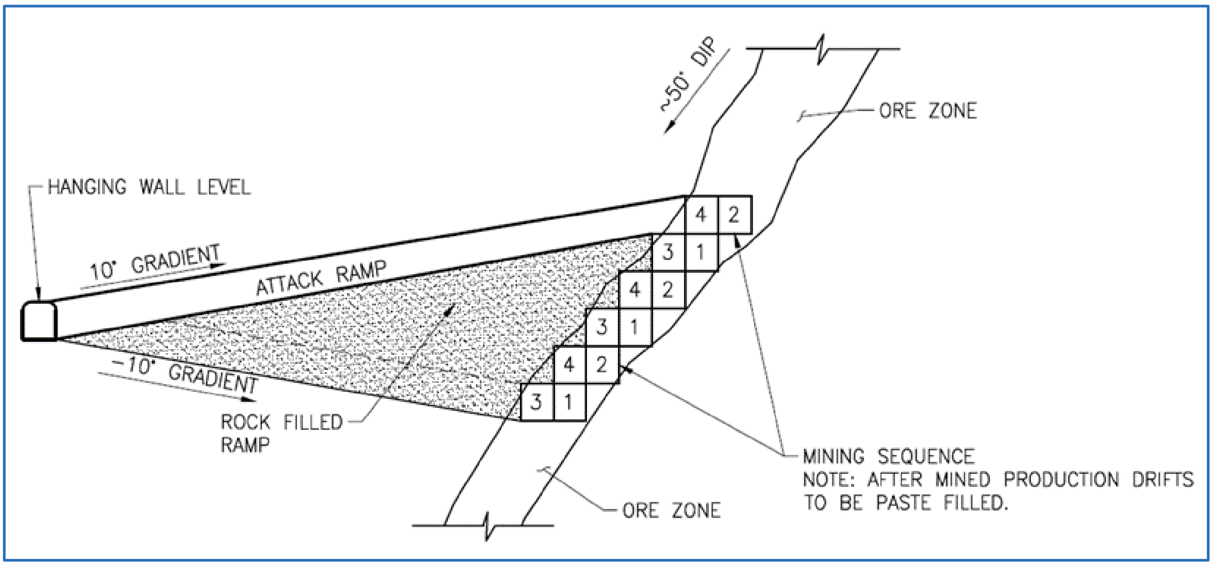
Table 16.24 Block Access Production Pillars and Connection Pillars

Depth (m)	Production Pillars (m)	Connection Pillars (m)
0–250	35	14
250–400	40	16
400–650	45	18
650–900	55	22
900–1200	65	26

16.3.3.6 Hanging Wall Accessed Drift-and-Fill

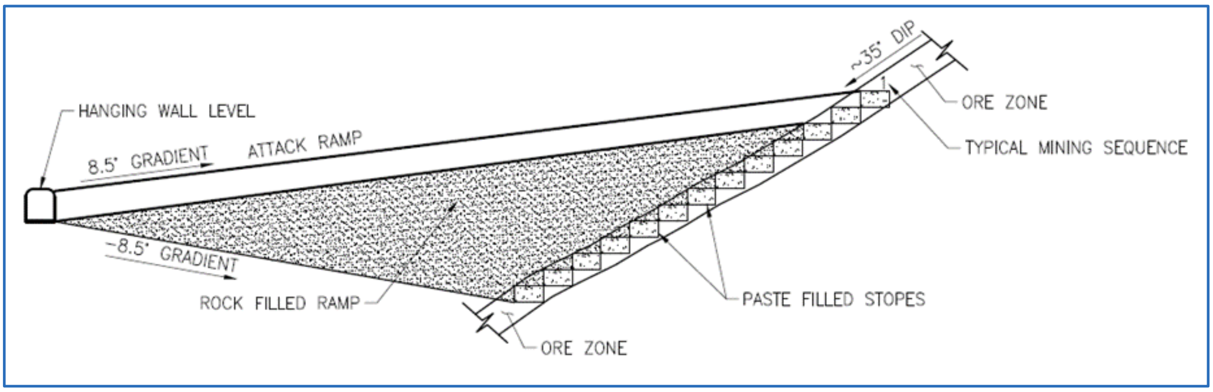
Two steep-dipping areas (>35°) occur at Kakula, the West area and the East area. In these areas, a typical production drift level is 36 m in height and accessed from development on the hanging wall. The west area production drift level comprises of six horizons of production drifts measuring 6.0 m H x 6.0 m W, as shown in Figure 16.13.

Figure 16.13 Section View of West Hanging Wall Accessed Drift-and-Fill Area



The East area production drift level comprises of 12 horizons of production drifts measuring 3.0 m H x 6.0 m W, as shown in Figure 16.14. Production drifts are accessed through attack ramps from the level development and developed on strike.

Figure 16.14 Section View of East Hanging Wall Access Drift-and-Fill Area

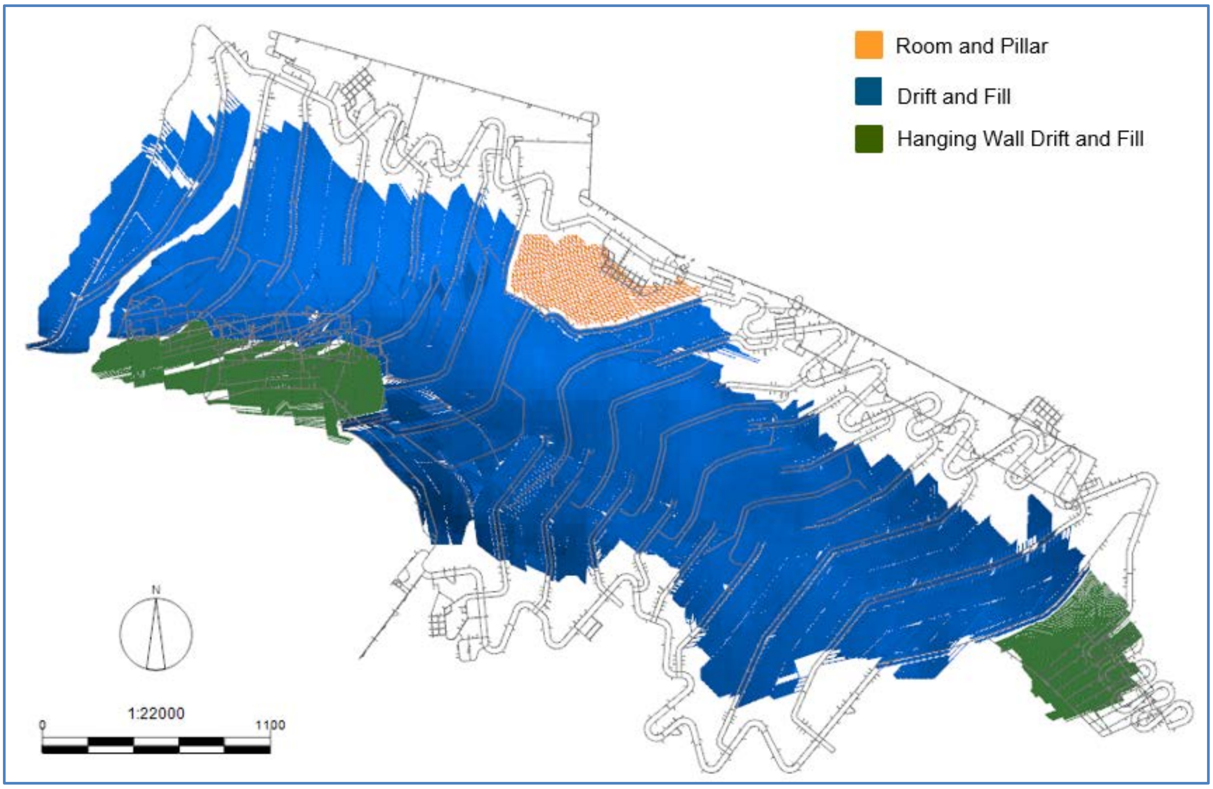


16.3.3.7 Mining Method Selection

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 99% of the targeted mineral resource) will be mined by drift-and-fill. Identified mining areas with a dip greater than 25° (about 13% of mining areas) will be mined using the hanging wall access drift-and-fill (HWAD&F) method. The Kakula deposit is illustrated in Figure 16.15, represented by mining method.

Controlled convergence room-and-pillar was excluded from use at Kakula as a mining method option due to several factors such as the orebody thickness and lower extraction ratios.

Figure 16.15 Mining Method by Location

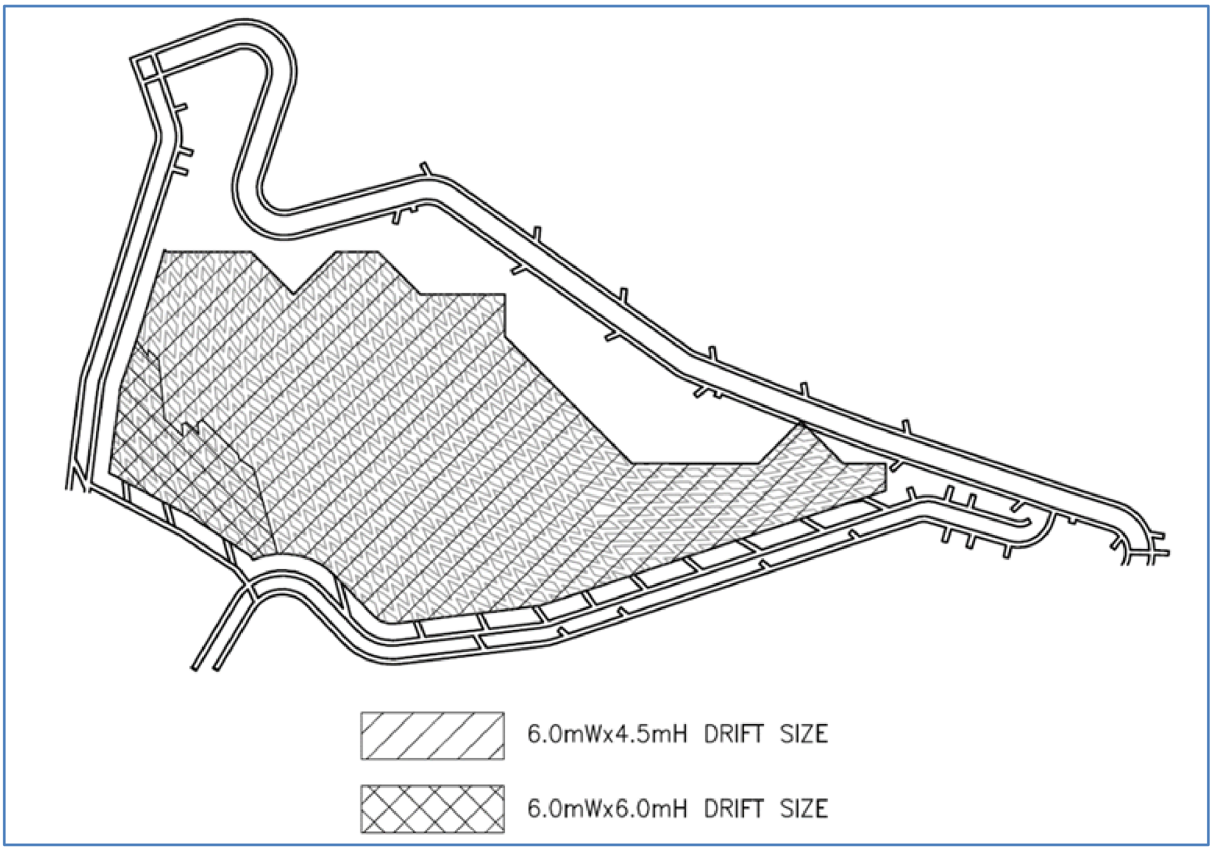


16.3.4 Mining Methods

16.3.4.1 Room-and-Pillar

The portion of the deposit to be mined by room-and-pillar is shown in Figure 16.16. Ore thickness ranges from 4.5–6 m and provide early ore production while drift-and-fill areas are being developed. Production development is 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 (1,360 kt).

Figure 16.16 Early Ore Room and Pillar Location



Every third main drift provides access to the room-and-pillar area from the connection drift. Crosscut drifts are driven from the main drifts at perpendicular angles. Drift heights for both mains and crosscuts are between 4.5 m x 6.0 m, depending on thickness of ore. The drifts have a maximum gradient of ±10°.

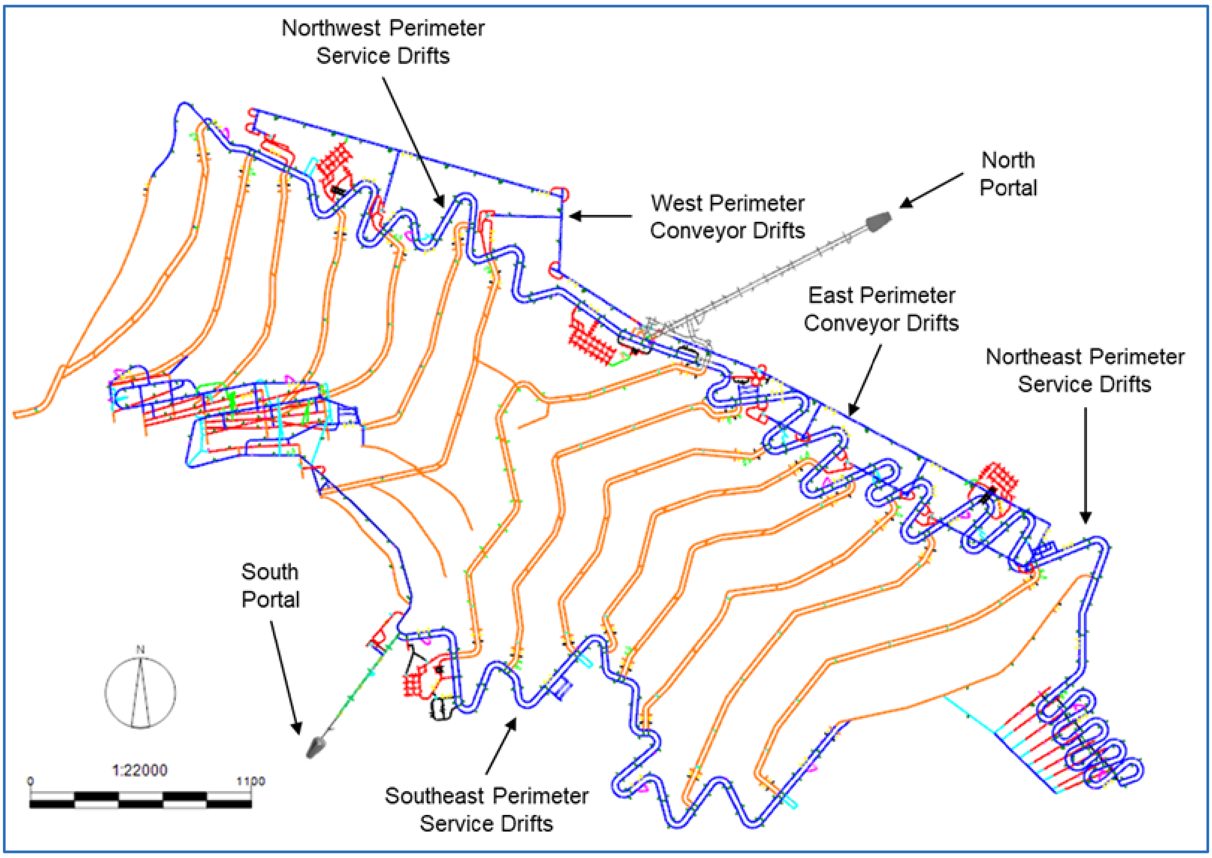
Main drifts run parallel to the strike of the panel for dips less than 20°, with cross-cut drifts running at an acute angle to the main drifts to ensure the grade of the cross-cuts remain less than 10°. Where the dip is greater than 20°, the main drifts are developed slightly off the strike to accommodate the acute angle between the main and the cross-cuts. This ensures 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.

16.3.4.2 Drift-and-Fill

Drift-and-fill mining is the primary method of extraction for the Kakula deposit. To establish the mining method, a pair of twin perimeter declines on the north and south of the targeted resource are driven at a defined offset. They are developed to the east and west extremities on the north side of the deposit, and to the east extremity on the south side of the deposit. A single perimeter decline is developed to the west extremity on the south side of the deposit. Twinned connection drifts are then developed across the targeted ore body, as shown in Figure 16.17.

Figure 16.17 Mine Development Framework



Twin connection drifts provide improved ventilation during the development stage, an alternative route of egress, and increased mining fronts during the production stage.

The distance between the connection drift centres depend on the dip and the depth of the zones. The connection drifts are driven on the footwall of the ore horizon and provide the framework: for defining the drift-and-fill mining blocks, accesses for equipment, and ventilation to the block access and production drifts.

Where the dip of the orebody is less than 20°, connection drifts are developed 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 are developed 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 are developed 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 divide the orebody into mining blocks. There is a mining block on either side of the connection drift, and each mining block contains three mining units.

Drift-and-fill mining and the associated connection drifts account for 108,615 kt of the Probable Mineral Reserve. This is comprised of 97,593 kt drift-and-fill mining and 11,022 kt from connection drifts.

The production drifts are 6 m W with a varying height of 3.0–6.0 m. There are two lifts in the areas where the ore height is 9.0 m or greater. The length of production headings is determined by the depth and dip of the ore body. Production headings contain temporary services required for advancement and backfilling.

The relative angle of the production drifts to the connection drift change based on the dip of the ore body. The maximum gradient for these drifts is 12°, and they are developed to a point half the distance between the connection drifts. For continuous mining, the primaries advance sequentially across the three mining units. As the primaries are completed, they are paste-filled in the same sequence. Once the primary drifts are completed and filled to the designed mining block width, the first set of secondary production drifts are available for extraction. This process continues with the tertiary drifts when the secondary drifts are completed and filled.

The extraction sequence of a mining block is illustrated in Figure 16.18.

Figure 16.18 Drift and Fill Extraction Sequence

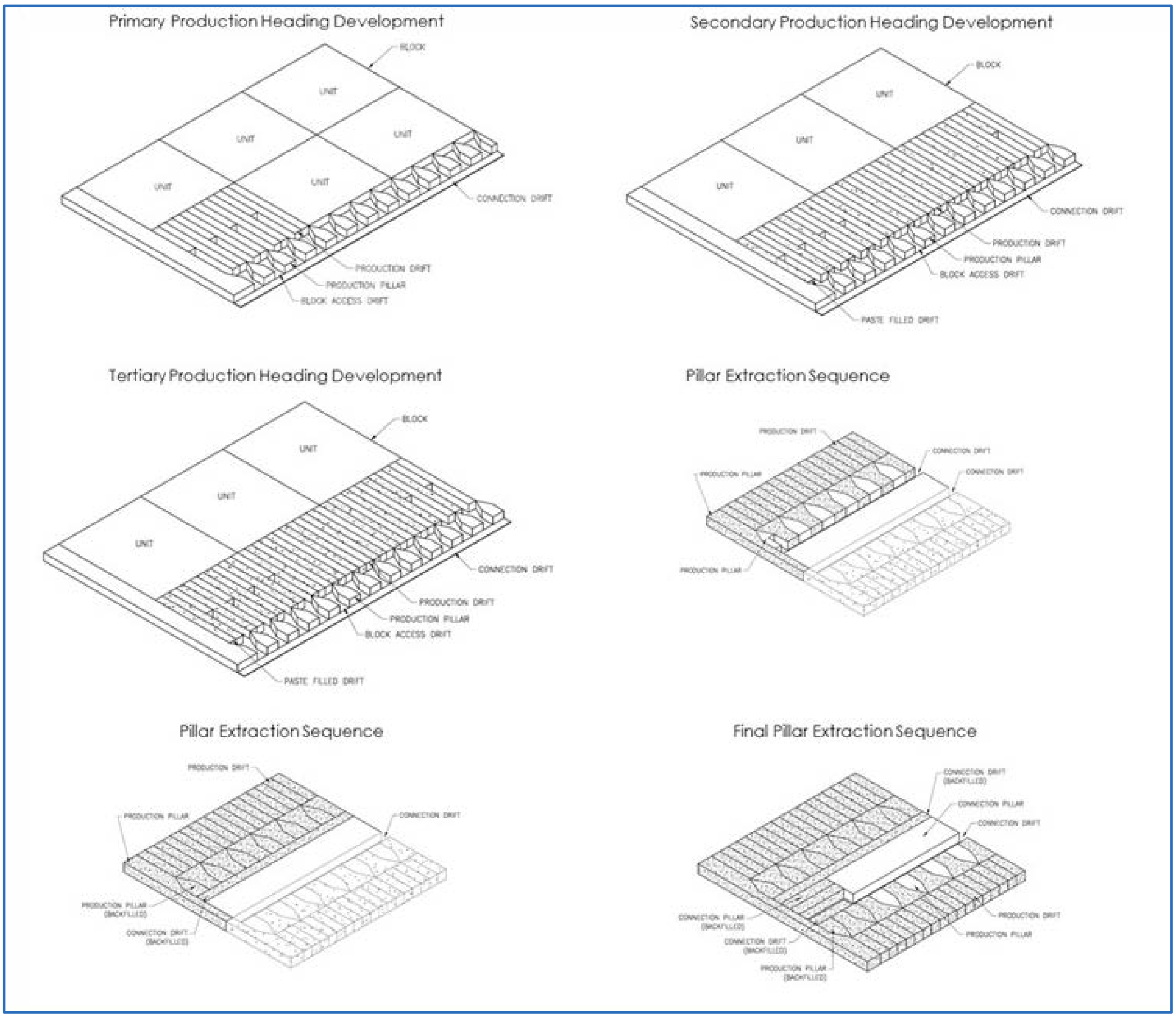


Figure by Stantec, 2019.

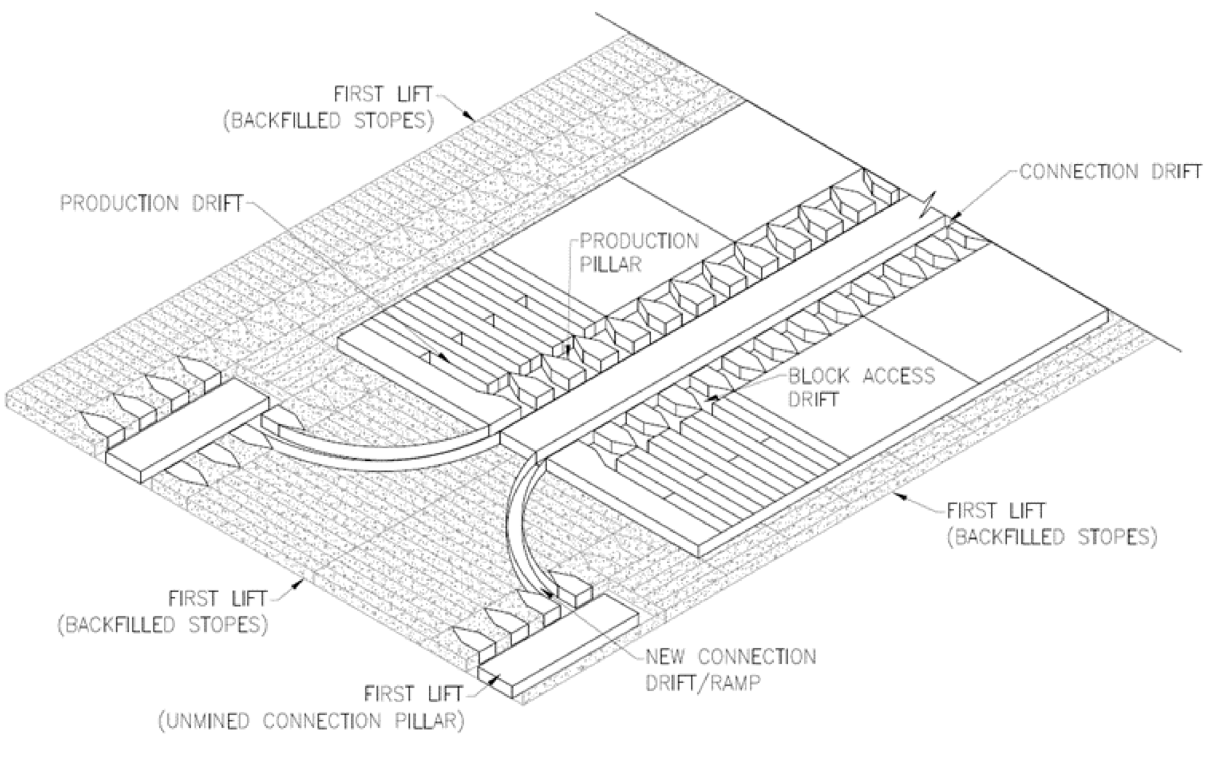
Paste fill is the primary backfilling strategy for drift-and-fill. The paste fill system includes a surface paste plant and a piping network connected to a series of boreholes that will deliver paste fill to drop points along the connection drifts, near the perimeter declines. Distribution pipes installed in the connection drifts deliver the paste fill to the production headings. After completion of each of the primary and secondary drifts, paste fill walls are constructed at the entrance to the drifts. Once the tertiary drifts are extracted, the final paste wall for that block will be constructed at the connection drift access. Since all production drifts are mined either up or down gradient, a strategy for filling and monitoring progress with no visual reference needs to be developed, along with a flushing and semi-continuous operating plan.

Pillar extraction is the final mining step to recovering the resource in the drift-and-fill method that is implemented at Kakula. Production pillars support the connection drift during production extraction. When the production headings in the mining unit are completely backfilled and cured, the production pillars are extracted. Production pillars and their corresponding connection drifts segments are filled and mined in sequence on retreat, typically from the south end to the north.

For ore thicknesses greater than 9.0 m, the first lift is first mined at 6.0 m H, and the remaining ore then evaluated to determine if it is suitable for mining with a 3.0–6.0 m H second lift. The second lift production is accessed from a second lift connection drift, developed from the existing first lift connection drifts. The drifts begin from the first lift elevation and ramp-up to the second level using the first lift production access as the start. The second lift connection drifts are offset midway between the sets of the first lift connection drifts, as shown in Figure 16.19. This arrangement satisfies geotechnical criteria and provides advantages in mining sequence, as well as better gains in ore production.

Ventilation loops are re-established by the new connection drives, as they are connected to both the north and south perimeter declines.

Figure 16.19 Second Lift Production



The second lift commences using the drift-and-fill method once the pillar extraction of the first lift is completed. The sequence of mining will be centre-out, per the first lift production sequence. When mining is completed on the second lift, the pillar extraction sequence of the second lift pillars can begin.

16.3.4.3 Hanging Wall Accessed Drift-and-Fill

The HWAD&F method is implemented in the two identified areas where the ore dips steeper than 25°. The perimeter twinned decline development is relocated to the hanging wall, over top of the ore body, at 5.5 m W x 6.0 m H, at a maximum of an 8.5°gradient, to access a production level every 36 m in height.

In the eastern area implementing the HWAD&F mining method, each level comprises of 12 horizons, as shown in Figure 16.13. Production drifts are driven on strike at 6.0 m W x 3.0 m H.

In the western area implementing the HWAD&F mining method, each level comprises of six horizons, and production drifts are driven on strike at 6.0 m W x 6.0 m H, as shown in Figure 16.14.

Production drifts are accessed via attack ramps from each production level. Fresh air is pulled into the hanging wall level and forced to the production headings via fans. The air is exhausted into 4.0 m diameter ventilation raises, located on the level. Declines and the hanging wall level are driven at 5.5 m W x 6.0 m H, and at a maximum gradient of 8.5°. Production drifts are accessed from each level via attack ramps and mined bottom-up, with two or less active drifts per attack ramp being mined at a given time.

After accessing each hanging wall level from the decline and establishing ventilation, the following mining sequence takes place.

1. Develop attack ramps on the first horizon of production drifts. There are two attack ramps per hanging wall level.
2. Mine the first horizon of production drifts on both directions from each attack ramp.
3. Paste fill the first horizon of production drifts.
4. Use paste or waste backfill to create attack ramps on the first horizon of production drifts to the height of the floor for the second production drifts horizon.

Due to the dip of the ore body, it may be possible to achieve multiple production horizons at the same time; this provides the flexibility to have multiple faces available at any given time for production needs. The subsequent mining sequence is to follow Steps 1–4 on each of the remaining horizons in a bottom-top manner.

16.3.4.4 Room-and-Pillar

The room-and-pillar mining method assumes a 100% extraction of the designed crosscuts and main drift shapes and leaves the pillars in situ. The overall extraction varies depending on depth.

16.3.4.5 Drift-and-Fill

The drift-and-fill method, including pillar extraction, assumes approximately 100% extraction of the mined shapes. The block sizes are based on using an 85% extraction of the ore in the first phase: the remaining 15% is from pillar extraction. Losses associated with mining and dips are applied using dilution and recovery factors. An additional 25% production pillar loss is applied to the standard mining block recoveries.

16.3.5 Dilution

The following criteria were used to develop the dilution strategy.

16.3.5.1 Dilution Grade

All production drift dilution grades are calculated by interrogating the feasibility Mineral Resource block model. The overbreak dilution varies by drift size and is included in the interrogated grade value for development drifts. In addition, all secondary, tertiary, and second lift production headings that contain paste dilution have a paste dilution component calculated within the schedule to produce final diluted grades. The paste fill tonnage has a zero-grade copper value.

16.3.5.2 Dilution Tonnes

Development Headings

Development headings have a flat back with 1.5 m radius arched corners. Back and wall dilution in all these drifts is assumed to have an average overbreak of 0.15 m with no floor overbreak. Some of the development headings include low-grade tonnes that are accounted for in production and development schedule tonnes and grade reporting.

Ore Development (Connection Drifts)

The ore development consists of the connection drifts and production access drifts that have a flat back and 1.5 m radius arched corners. An average overbreak is assumed to be 0.15 m in the back and 0.15 m for each of the walls.

Room-and-Pillar

Room-and-pillar mining only includes production from the main and crosscut drifts with flat backs. For back dilution, overbreak is assumed to average 0.15 m. No dilution from the walls is considered, since the pillar width must be maintained and any dilution would be consistent with the ore grade of the drift. Controlled blasting practices are required to ensure that the walls are broken to the design width and that the flat backs are maintained. The footwall dilution is internal planned dilution and is based on the dip and thickness of the production panel shape.

Drift-and-Fill

Drift-and-fill will be the primary method of ore extraction. First pass will include the connection drifts, production access drifts, and the primary, secondary, and tertiary headings in the production blocks. The second pass will be pillar extraction.

The dilution percentages vary based on ore dip and thickness as well as the extraction sequence.

The height of the production areas is determined by the thickness and dip of the ore body. The hanging wall is followed with a shanty back to reduce the total internal dilution. In areas where a second lift is be mined over the first, the first drift will have a flat back.

Primary drifts have only back dilution, since any overbreak into the walls is within the planned excavation. Secondary drifts have hanging wall dilution and paste fill dilution from one wall. Overbreak in the rock rib is in ore and not included as dilution. Tertiary drifts have hanging wall dilution and paste fill dilution from both walls. Additional overbreak is expected in tertiary drifts due to the worsening ground conditions.

Pillar extraction consists of the removal of both the production access pillars and the centre connection pillar. The dilution is anticipated to be an average of 0.15 m into the hanging wall and adjacent paste fill repeated for every step of the extraction pillar sequence.

Hanging Wall Accessed Drift-and-Fill

The Eastern HWAD&F dilution comes from overbreak of 0.15 m on each wall and the back. There is 0.15 m paste dilution from the floor.

The Western HWAD&F dilution for the primary drift is 0.15 m overbreak in the back. The dilution for secondary drifts includes 0.15 m paste dilution in one wall as well as back overbreak. Each lift also has paste floor dilution of 0.15 m.

16.3.5.3 Recovery Factors

The mining recovery includes allowances for equipment limitations, heading shapes, heading strike and dip angles, ore re-handling, and operator skill.

For all development, the recovery is 98.4%. Lost tonnage is due to the corners of the drift and ore that settle into floor irregularities; 0.15 m of rock material is lost on the floor.

The mining methods selected for Kakula are development-intensive and have recoveries similar to ore development. The average height of the material left is estimated to be 0.15 m where the mining is single lift. For the second lift of a double lift, ore recovery is assumed at 100% since all ore will be taken with the 0.15 m of backfill dilution.

In the HWAD&F west zone, the primary development drift at each mining lift has some loss of ore in the portion of the drift that is not over the top of the previous level. This loss of tonnage is offset by the tonnes of paste loaded out of the floor on the paste portion of the drift.

The pillar extraction phase requires each round to be mined before advancing. In general, the recovery is similar in all development headings at 98.4%. The final drift in each section of the pillar extraction assumes a 66% recovery.

16.3.6 Backfill

Paste fill will be the primary backfilling strategy for the Kakula Mine. The paste fill system will include a surface paste plant and a piping network connected to a series of boreholes that will deliver paste fill to drop points adjacent to the connection drifts, near the north perimeter declines. Distribution pipes installed in the connection drifts will then deliver the paste fill to the production areas.

Cemented sand fill will be used for approximately 10 months until the mill is producing sufficient tailings to supply the paste plant. Local surface sand will be mined and processed through the paste plant as cemented sand fill until the transition to paste fill in October 2021.

16.3.6.1 Material properties

Backfill will be produced using both Dambo sands and process plant tailings. Dambo sands is much coarser than full plant tailings and will be used until the process plant commences producing tailing. The sands may also be used with process plant tailing to reduce the binder content required.

The material properties and the particle size distribution in summarised in Figure 16.20.

Figure 16.20 Backfill Particle Size Distributions

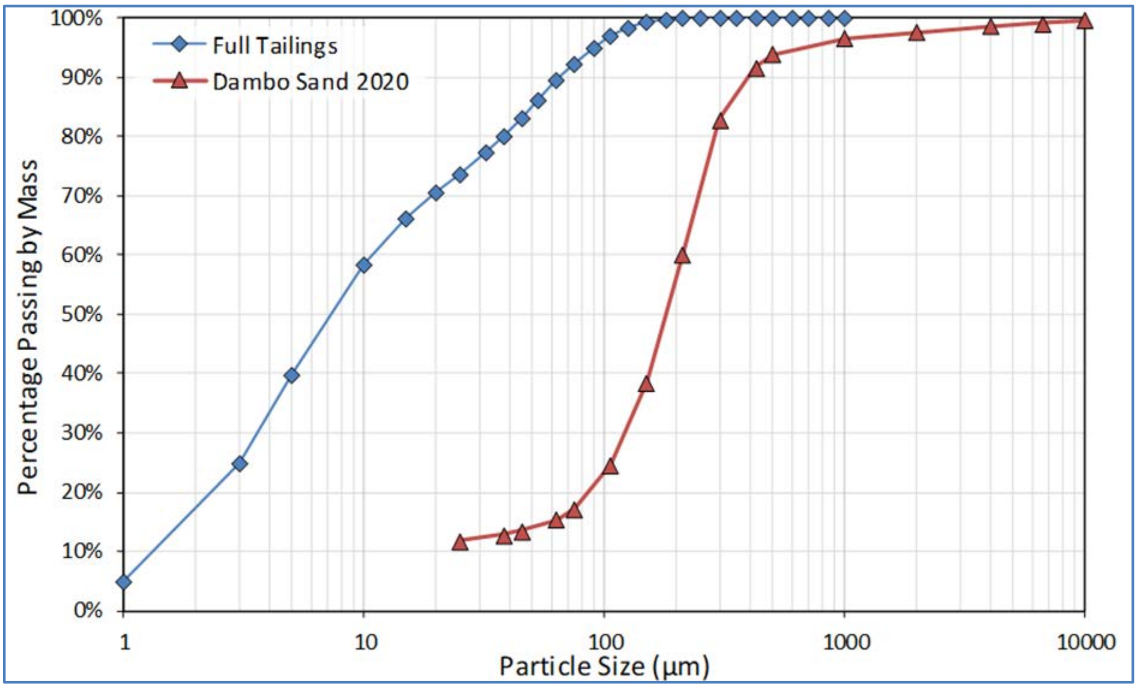


Figure by Paterson & Cooke 2020

The process options and modular design of the plant allow several blends of materials to be produced. Test work was conducted on various full tailings and Dambo sands recipes over a range of mass concentrations, water, and binder contents. The water:binder ratio verses strength correlations for 28 days curing are shown in Figure 16.21.

Figure 16.21 28 Day Unconfined Compressive Strength Correlations

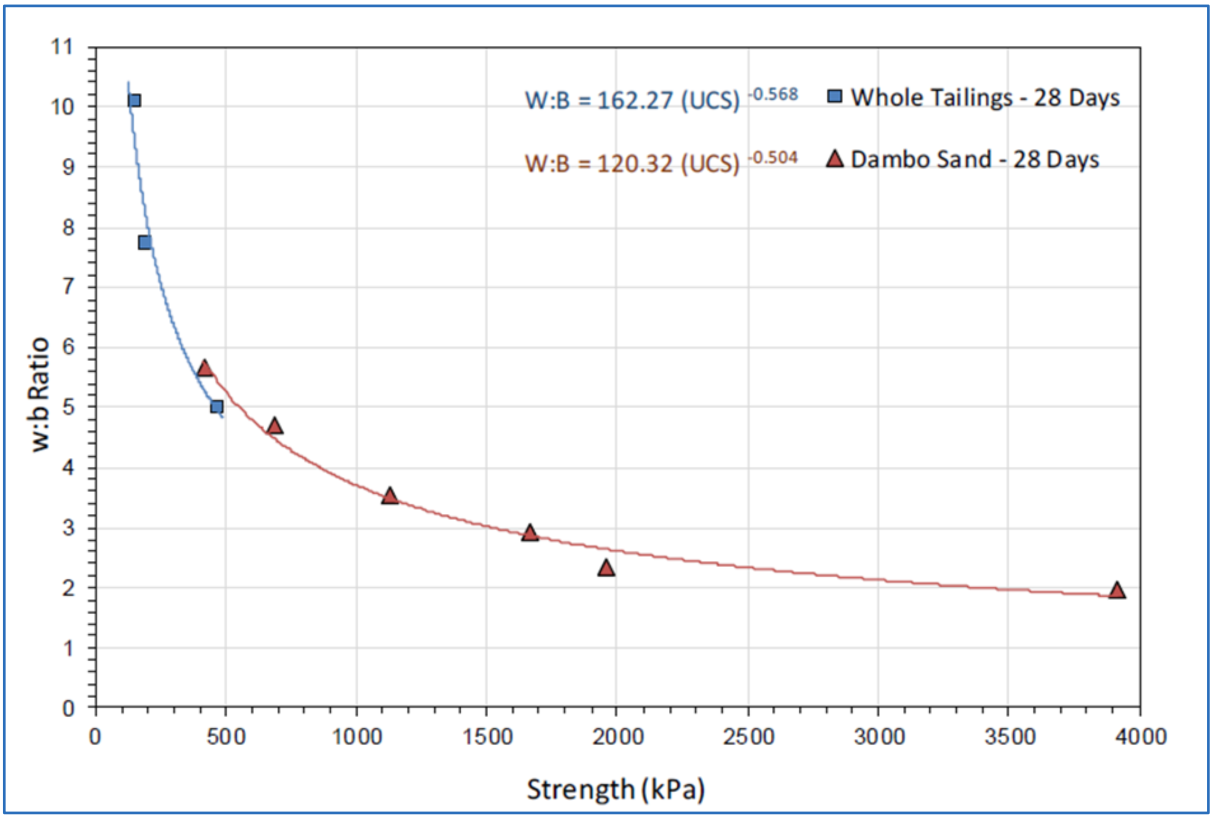


Figure by Paterson & Cooke 2020

16.3.6.2 Backfill Plant

The paste backfill plant is located to the south of the processing plant on the north side of the orebody above the north perimeter drift.

The paste plant facility consists of two multilevel buildings, a sand system area, and a binder storage area. The plant components for each module consist of filter feed tanks, vacuum disc filters, filter cake conveyors, sand surge bins, sand feeder conveyors, binder silos, paste mixer, paste hopper, and positive displacement pumps.

The paste plant is designed as four modules. The process flow schematic is shown in Figure 16.22.

Figure 16.22 Process Flow Schematic for Backfill Plant

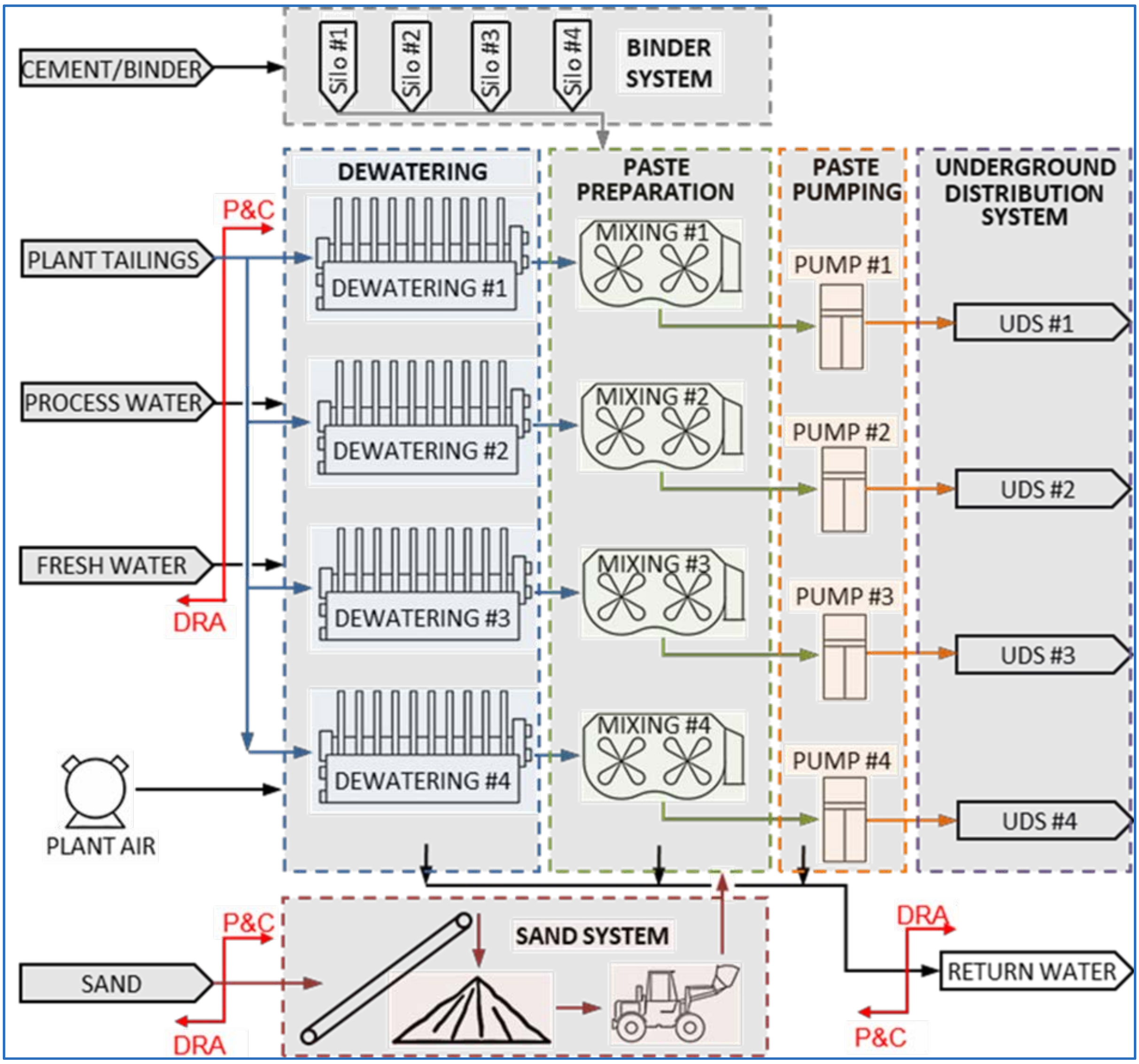


Figure by Paterson & Cooke 2020

16.3.6.3 Backfill Distribution System

A schematic of the paste plant location and surface pipe routes, boreholes, and underground connection drifts is shown in Figure 16.23.

Reticulation pipe diameter is nominally 200 mm. The pipe specification varies depending on four categories of pressure rating: high pressure surface and underground, boreholes, low pressure underground, and in stope.

There are four high pressure pipelines (one for each module) east and west from the paste plant on surface. These lines will be extended to reach additional boreholes as the mine is developed.

Figure 16.23 Surface and Underground Backfill Distribution Schematic

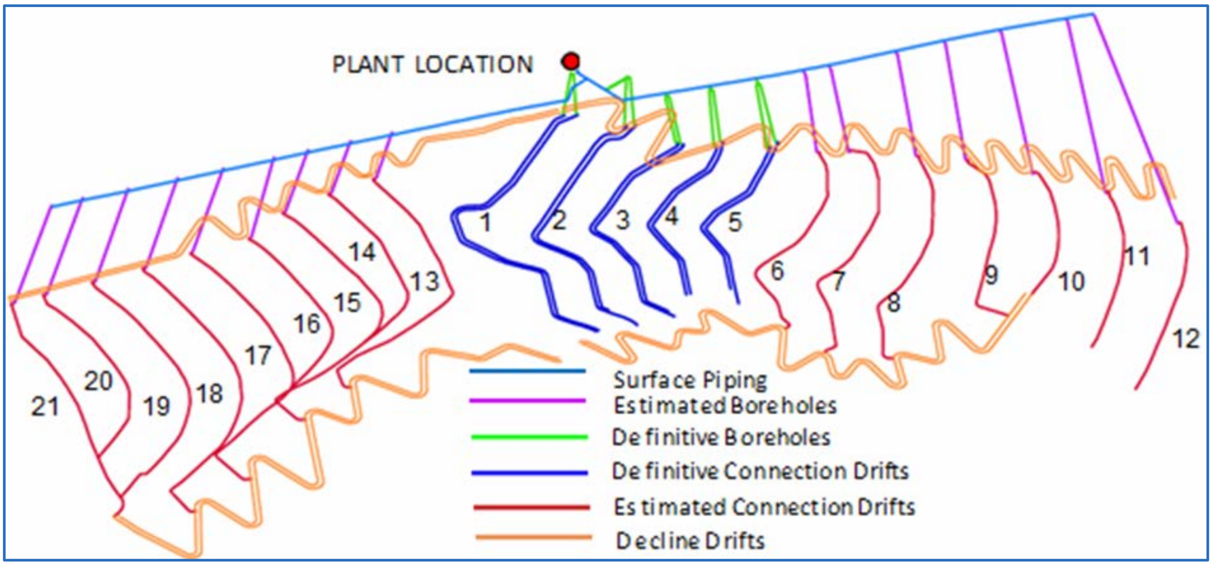


Figure by Paterson & Cooke 2020

Positive displacement pumps are used to pump the paste material to the borehole collars. The positive displacement pump sizing was based on limits in commercially available pump size and associated available discharge pressure Table 16.25.

Table 16.25 Positive Displacement Pump Specification

Specification	Description
Pump type	Hydraulically driven positive displacement piston pump
Flow rate	Operating range: 130–170 m^3/h Nominal flow rate: 150 m^3/h
Slurry density	Operating range: 1.63–1.96 t/m^3
Pumping pressure	Mean pump pressure: 14 MPa
Hydraulic power per pump	Maximum: 725 kW
Absorbed power per pump	Maximum: 805 kW @ 90% efficiency
Installed power per pump	2 x 450 kW motors (hydraulic power pack)

The connection drifts have a twin drive arrangement with each drift being supplied by a surface borehole. The boreholes from surface will be designed at 70–80° dip, lined with steel pipe, and exit into mined backfill cuddies on each connection drift. At each borehole location, each of the four pipelines has provision to swing over to both boreholes allowing all paste plant modules to deliver paste to all connection drifts.

Steel pipe hung from the backs will distribute the paste from the boreholes along each drift. Various pipe hangers, guides, and anchor supports are used to support the pipe depending on the location and duty. The distribution system also includes rupture discs and emergency dump tees to protect the pipeline from over pressurisation or to allow dumping of backfill material during a blockage event.

When a production area becomes available for backfilling an elbow is connected into the connection drift pipe to divert the backfill flow. This diversion elbow piping assembly includes a valve to divert flush water, and a transition from steel pipe to HDPE (high density polyethylene) for final delivery into production areas.

16.3.6.4 Backfill Schedule

The underground paste distribution system is based on the Life-of-mine requirements of the drift-and-fill mining.

Table 16.26 and Figure 16.24 detail the fill requirements by year.

Table 16.26 Fill Requirements by Year

Cemented paste fill ('000 t)									
Year	**Total**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**
Kt	69367	–	374	1,625	2,192	3,434	3,487	3,750	3,931
Year	**2028**	**2029**	**2030**	**2031**	**2032**	**2033**	**2034**	**2035**	**2036**
Kt	3,810	4,109	3,712	3,835	3,988	4,079	3,993	3,913	4,200
Year	**2037**	**2038**	**2039**	**2040**	**2041**	**2042**	**2043**	**2044**	**2045**
Kt	4,138	4,314	2,936	1,991	1,556	–	–	–	–
Cemented sand fill ('000 t)									
Year	**Total**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**
Kt	874	74	800	–	–	–	–	–	–

Figure 16.24 Backfill Schedule

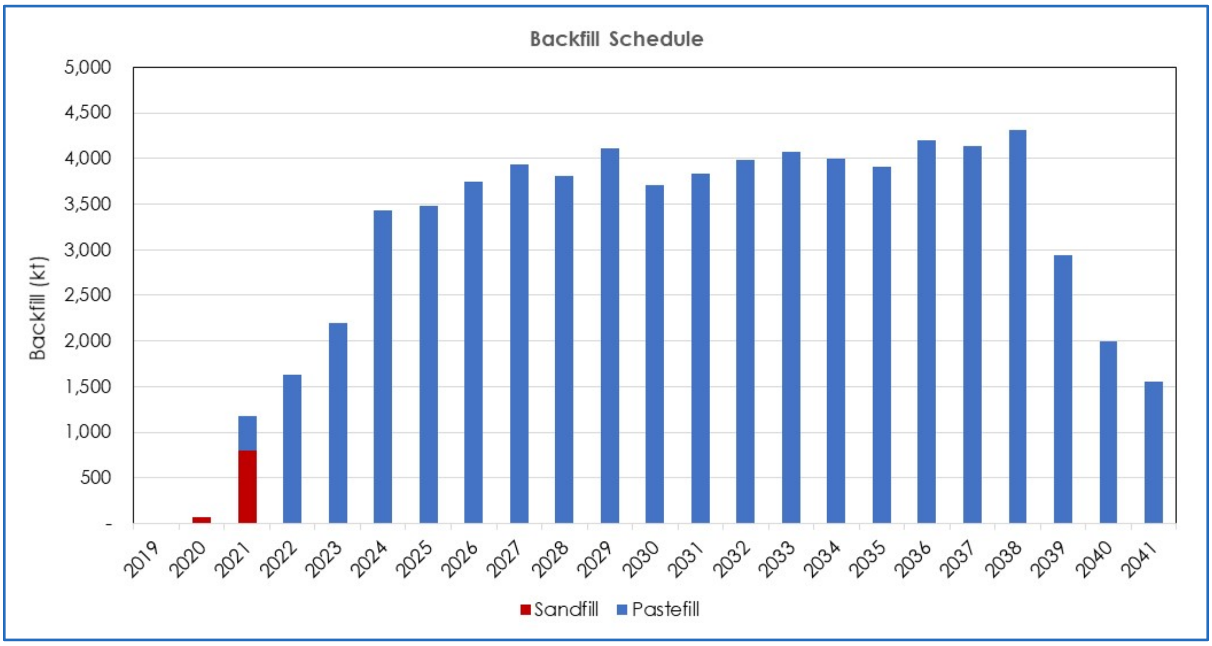


Figure by Stantec, 2020.

16.3.6.5 Sand Mining

The Kamoa-Kakula project will use sand (mined from the planned TSF footprint) as backfill material until tailings from the TSF becomes available. This sand mining operation will mine approximately 415,000 m³ of sand for backfill over a twelve-month period and will build a stockpile that can be used during periods when tailings are not available for paste fill. An additional 88,000 m³ will be mined for a further two months to ensure that the backfill sand stockpile remains full at the end of the twelve-month backfill period.

Figure 16.25 Sand Mining Target Area in the TSF Footprint

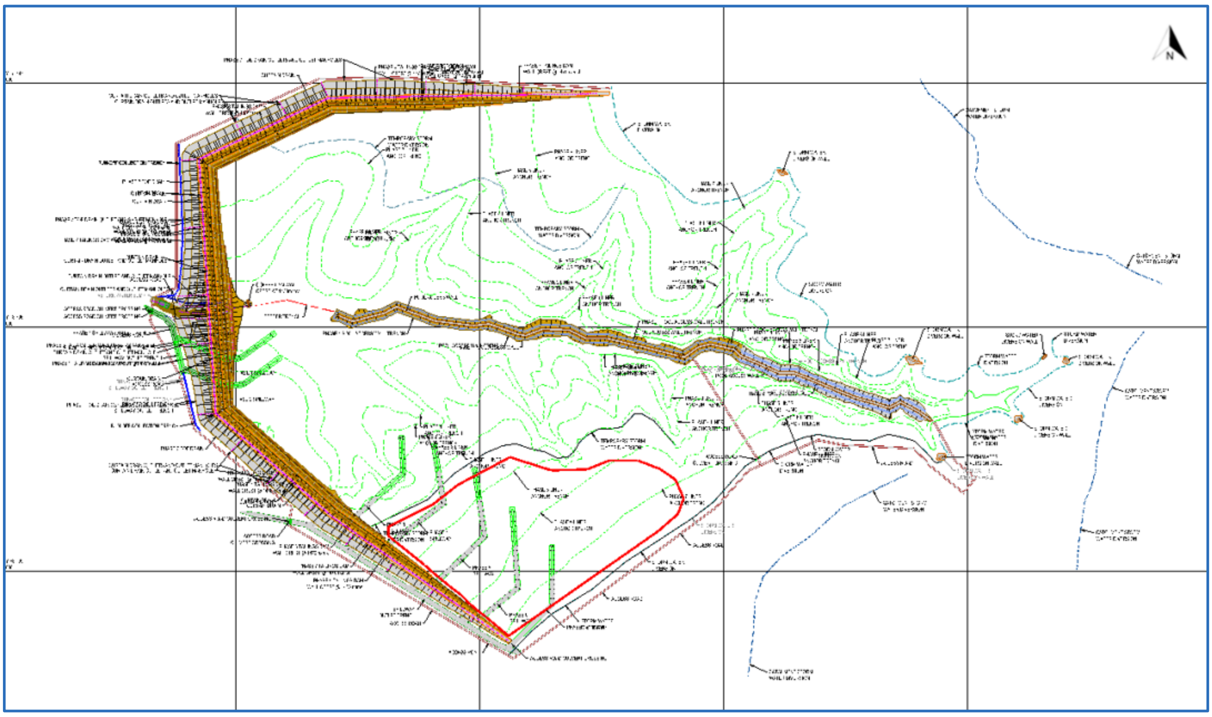


Figure by DRA, 2020.

The mine design scope includes the following design considerations:

- Elementary modelling of material surfaces is based on auger hole logs
- Dewatering entails limiting use of pumping
- Sand mine design footprint is enough in meeting initial BF and possible future expansion requirements
- The capability of available excavators on site were considered in the mine design

Over a twelve-month period, 415 km^3 of sand must be produced for BF as illustrated in Table 16.27

Table 16.27 Sand Mining Production Schedule

	Unit	Month 1	Month 2	Month 3	Month 4	Month 5	Month 6	Month 7	Month 8	Month 9	Month 10	Month 11	Month 12	Month 13	Month 14	Total
Bush clear																
Area	m^2	24,799	20,396	21,405	21,455	21,364	30,599	30,581	30,659	30,591	30,867	30,631	30,542	12,465	–	336,354
Thickness	m	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	–	0.50
Volume	m^3	12,000	10,000	10,500	10,500	10,500	15,000	15,000	15,000	15,000	15,000	15,000	15,000	6,094	–	164,594
Topsoil																
Area	m^2	15,088	20,985	21,746	7,277	5,197	24,998	14,689	14,577	21,527	13,965	25,506	27,474	25,941	22,466	261,434
Thickness	m	0.38	0.46	0.46	0.19	0.18	0.36	0.61	1.05	0.70	1.07	0.58	0.55	0.59	0.66	0.58
Volume	m^3	5,681	9,470	10,093	1,352	907	8,884	8,887	15,000	15,000	15,000	15,000	15,000	15,000	15,000	150,274
Sand																
Area	m^2	–	17,879	17,429	16,658	17,152	14,529	18,736	23,803	23,507	24,699	26,492	30,965	24,182	27,901	283,933
Thickness	m	–	1.86	1.82	1.88	1.80	2.10	2.37	1.90	1.98	1.82	1.71	1.40	1.90	1.64	1.83
Volume	m^3	–	29,974	29,974	30,182	30,182	30,182	44,165	44,165	44,165	44,165	44,165	44,165	44,165	44,165	503,812
Clay																
Area	m^2	–	759	234	226	162	685	917	–	390	–	–	110	–	–	3,483
Thickness	m	–	0.03	0.05	–	–	0.01	0.01	–	0.03	–	–	0.01	–	–	0.02
Volume	m^3	–	23	11	–	1	9	11	–	12	–	–	1	–	–	69
Total Material																
Volume	m^3	17,681	49,467	50,578	42,034	41,589	54,075	68,063	74,165	74,177	74,165	74,165	74,166	65,259	59,165	818,749

16.3.7 Mine Access Designs

Box-Cuts

There are two box-cuts developed for access to the underground workings. The northern box-cut incorporates two portals and the southern box-cut has a single portal.

Main Declines

The deposit is accessed via twinned declines on the north side, and via a single decline on the south side.

One of the north declines is the primary mine service access, and the other decline is a conveyor haulage drift. The service decline has dimensions of 5.5 m W x 6.0 m H, and the conveyor decline is 7.0 m W x 6.0 m H. Both northern declines have an 8.5° gradient. Development of the access declines has been completed.

The main conveyor decline is wide enough to accommodate the conveyor as well as large mobile equipment, such as a Load haul dumper (LHD) or truck, while still maintaining pedestrian access. Travel direction in the declines is uphill in the main conveyor decline and downhill in the main service decline. Approximately 1,030 m from the portal, twin access drifts are developed off the main service decline to the perimeter declines. These drifts also provide access to the main workshop and the initial truck tip area, as show in Figure 16.26.

Figure 16.26 Main Access Development

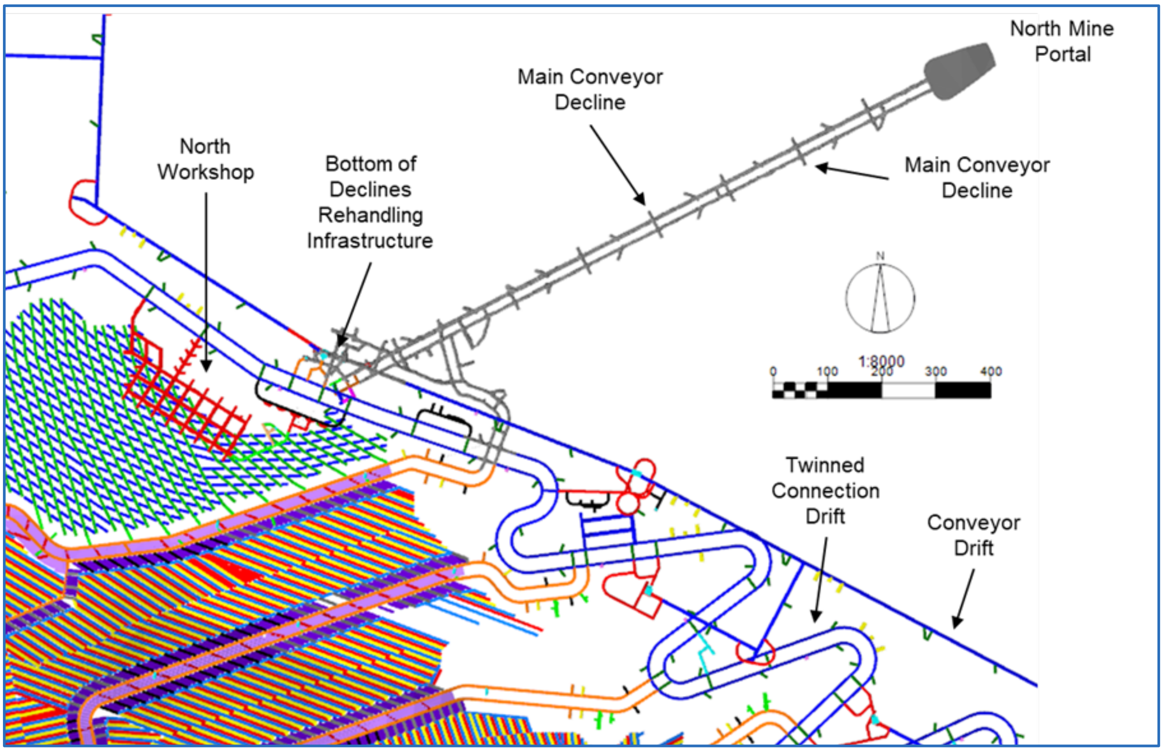


Figure by Stantec, 2020.

The south decline is 7.5 m W x 6.0 m H and is 640.0 m L at a 9.5° gradient. The development of this decline is in progress. It will target the perimeter decline elevation that will provide access to the southeast and southwest dual perimeter declines, as shown in Figure 16.27. This decline will provide ventilation and early access to development across the ore body.

Figure 16.27 South Access Development

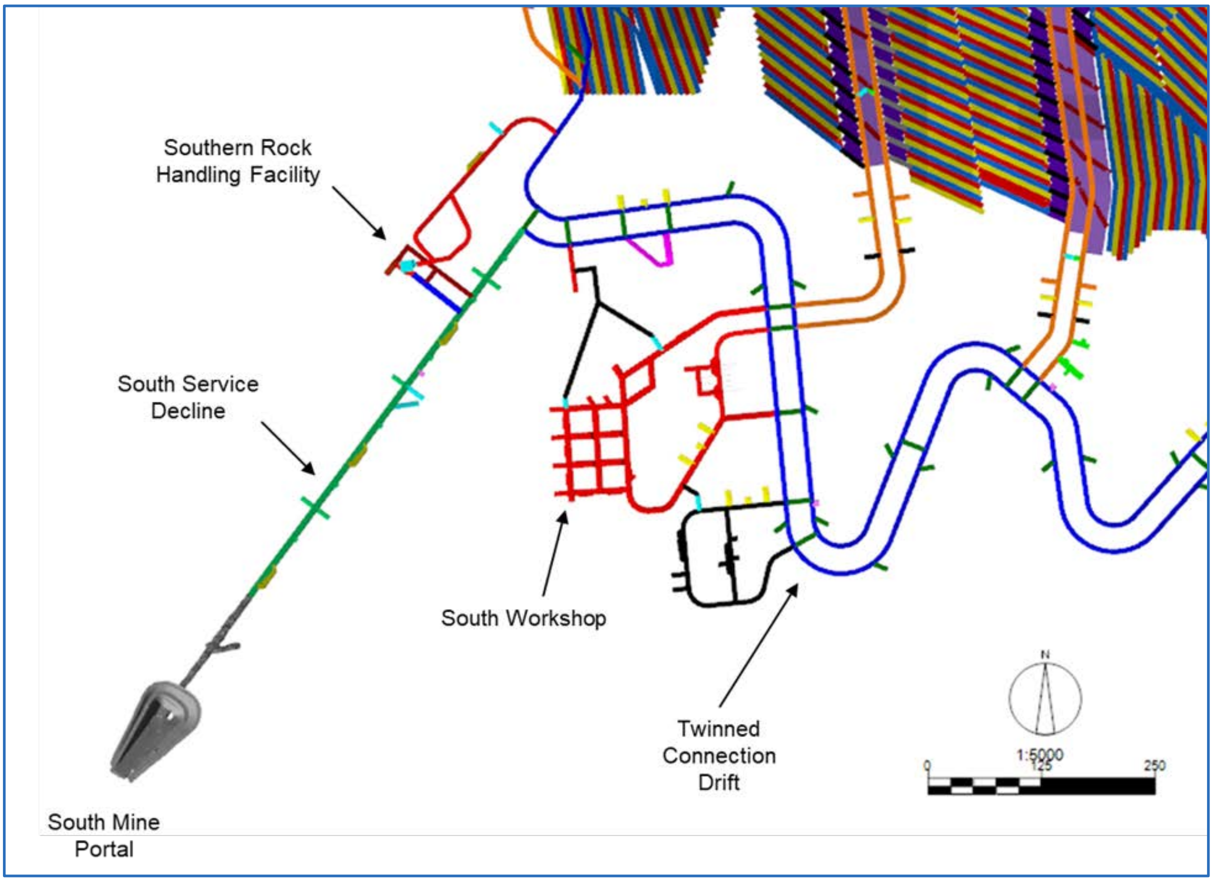


Figure by Stantec, 2020.

Perimeter Service and Conveyor Drifts

From the bottom of the north and south declines, a pair of 5.5 m W x 6.0 m H perimeter service drifts will be driven to the east and west extremities of the deposit. This development will serve as the primary accesses to the production areas and underground infrastructure. These perimeter drifts will also 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 on the outside of the north perimeter declines and will converge at the main conveyor decline. Conveyor drifts will be 7.0 m W x 6.0 m H and will be driven at a maximum gradient of 10.0°.

Mining Areas

The initial connection drifts between the north and south declines serve as the primary access to the room-and-pillar mining area, along with the adjacent perimeter declines.

For drift-and-fill mining, connection drifts will be developed between the north and south perimeter declines. These will serve as the main accesses to the production blocks. Connection drifts between the north and south perimeter declines will provide access and ventilation to the planned mining areas.

16.3.8 Mine Development and Production Schedules

The development schedule focuses on establishing mine services and support infrastructure to set up the initial production mining areas and to ramp up to 6.0 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 are in the following phases.

- Phase 1: Development of the north declines (completed early 2019).
- Phase 2: Development of the south decline and production rock handling facilities (completed 2019).
- Phase 3: Start room-and-pillar mining (commenced Q2’20).
- Phase 4: Development of initial twinned connection drifts to establish primary ventilation circuit and start of drift-and-fill mining (Q4’20).
- Phase 5: Initial full production (2024).
- Phase 6: Life-of-mine.

Table 16.28 summarises the LOM development and production results.

Table 16.28 Life-of-Mine Development and Production Summary

Waste Development	
Lateral (m)	91,284
Lateral (t)	9,124,380
Mass Excavation Lateral Equivalent (m)	4,413
Mass Excavation (t)	508,382
Vertical (m)	35,578
Vertical (t)	480,324
Low-grade (m)	52,225
Low-grade (t)	5,200,043
Low Grade NSR ($/t)	57.04
Low Grade TCu (%)	1.28
Total Waste Development	
Total (m)	183,499
Total (t)	15,313,129
Production by Mining Method	
Ore Development (m)	105,217
Ore Development (t)	11,022,401
Room-and-Pillar (m)	17,053
Room-and-Pillar (t)	1,359,646
Drift-and-Fill (m)	938,229
Drift-and-Fill (t)	80,423,002
Pillar Extraction (m)	196,659
Pillar Extraction (t)	17,170,014
Total Ore Production	
Total Development (m)	105,217
Total Production (m)	1,151,941
Total Tonnes (t)	109,975,063
Diluted Grade	
NSR ($/t)	243.18
TCu (%)	5.22
S (%)	1.45
As (%)	0.00
Fe (%)	4.77
Density (t/m^3)	2.82

1. Notes:Vertical development includes boreholes.
2. Low-grade cut-off NSR grade — US$20.
3. Stope shapes designed on an NSR cut-off value of US$100.

The following conditions were used in developing the LOM schedule:

- Proximity to the main accesses and early development.
- High TCu grade 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 TCu grade profile during ramp-up and full production. Low-grade material was classified by NSR values, which range from $20–$100.

16.3.8.1 Productivity Rates

Effective Operating Hours

The effective operating hours per shift are summarised in Table 16.29 and represent the time a crew is expected to spend actively working (effective working time). This was estimated to be 8 h/shift. The effective working time per shift was applied throughout the first principles rate calculations except for borehole installation, raise boring, and underground facility construction. These were contractor-supplied based on the contractor's own daily shift schedules and were included in the schedules and productivity estimates for these activities.

Table 16.29 Shift Rotations and Effective Operating Hours Calculations

Shift Cycle	Calculations
Days per Year	360 days
Crew Rotation (days)	4 on / 4 off
Number of Crews in Rotation	4
Shifts per Day	2
Shift Duration	12 h
Traveling Time – In	19.50 min / 0.32 h
Traveling Time – Out	19.50 min / 0.32 h
Lunch	60 min / 1 h
Pre-Shift Safety Meeting and Pre-Shift Inspections	45 min / 0.75 h
Actual Face Time per Shift	577 min / 9.61 h
Actual Face Time per Day	1153 min / 19.21 h
Effective Working Time per Hour (50 min/h)	83%
Effective Face Time per Shift	8.0 h
Effective Face Time per Day	16.0 h

Horizontal Development

For primary development, the rates were calculated using first principles. Cycle inputs were obtained from various sources (such as Kakula historic rates, OEM, external consultants, specialists, Stantec 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 16.30.

Table 16.30 Development Rates

Description*	Single-Heading Performance (m/day)	Double-Heading Performance (m/day)
Perimeter service drift 5.5 m W x 6 m H – Semi-arch	3.80	N/A
Perimeter conveyor drift 7 m W x 6 m H – Semi-arch	3.80	N/A
Connection Drift 6 m W x 6 m H – Semi-arch	3.80	5.20
Mass Excavation	3.80	N/A
South Decline – 7.5 m W x 6.0 m H – with Passing Bays (Contractor Data)	4.10	N/A

Note: Double heading rates were used for re-muck bays, dams, and other bay development from drifts where it was appropriate.

Vertical Development

Raise boring rates used in the Project schedule are from contractor experience or from recent contractor quotations. Boreholes are raisebore pilot holes. All ventilation shafts and raises are assumed to be raisebored and include allowances for ground support.

Production rates

Production rates for the mining methods were estimated by identifying the critical activity in the mining cycle and using this to estimate the total rate of the cycle. The mining methods are all development style activities and so the rate was calculated and assigned in the schedule in terms of development metres. The rate in ore tonnes per day is highly variable as it is an outcome of the estimated development advance rate, combined with ore thickness, dilution parameters, and ore density.

The development advance rates per crew for each mining method are summarised in Table 16.31.

For room-and pillar mining, the average ground support installation time, was combined with face drilling, and loading to determine the critical activity.

For drift and fill production rates, development rates for primary, secondary, and tertiary drift-and-fill drifts were combined with paste filling and end-of-shift blasting restrictions in a block configuration, and then analysed to determine the net block production rate for use in the schedule. The maximum four blasts per day performance was reduced by a factor of 26 production days out of 31 (83.7%) to allow for missed or delayed cycles within a round from various interference factors. Paste fill barricade construction and placement was not considered in the cycle calculations as it will be completed off critical task.

Table 16.31 Production Rates

Description	Single Heading performance (m/day)	Critical Activity Performance (m/day)
Room-and-pillar	3.80	7.60
Drift-and-fill	3.19	12.75
Block access	3.19	12.75
HWAD&F	3.80	7.6

For HWAD&F, the first principles individual production rates used end-of-shift blasting restrictions in a block configuration to determine the net block production rate (t/day). This rate was then used in developing the production schedule.

For an individual attack ramp that has two faces available, combined with end-of-shift blasting, the maximum sustainable production performance was two rounds per day per attack ramp. There was no performance reduction due to the complexity of the cycle. Paste fill barricade construction and placement impact the overall schedule as a result, the paste fill and associated activities were scheduled as a separate task and not included in the production cycle.

16.3.8.2 Development Schedules

Preproduction and Production Ramp-up Development

The preproduction and ramp-up period started in June 2019 and is scheduled to end Q4'21. Key development activities to be completed in this period include the following:

- South decline.
- Initial ventilation raises VS-1 and VS-2.
- Rock handling infrastructure around bins and conveyor loading area.
- Centre connection drifts between north perimeter service drift to south perimeter service drift.
- Construction and development of other critical infrastructure such as: conveyors, initial truck tip points, return water pumping, refuge stations, and underground workshops.

The ramp-up period to full production concludes at the end of 2023. Figure 16.28 illustrates the development metres associated with the pre-production and ramp-up activities.

Figure 16.28 Preproduction / Ramp-up Development Schedule

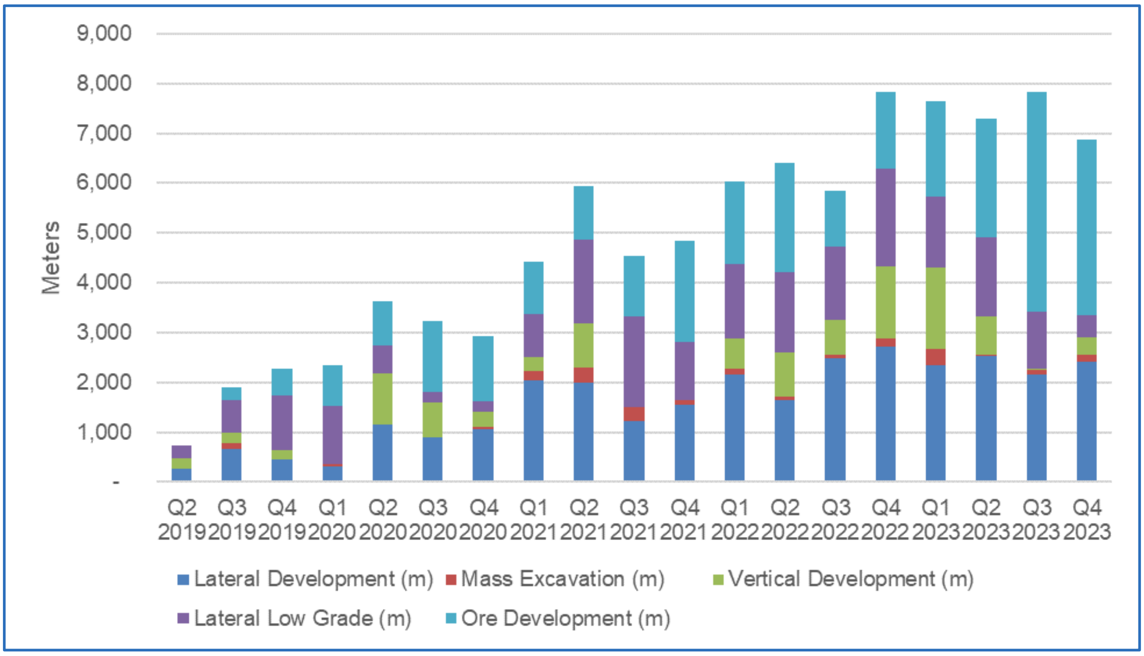


Figure by Stantec, 2020.

Development schedule

The life-of-mine development schedule beyond the initial preproduction period and ramp-up targets the areas required after 2023 to support the LOM plan. This includes excavating the perimeter service drifts, conveyor drifts, and key infrastructure associated with truck tips, ventilation, and maintenance facilities in advance of production areas. Figure 16.29 illustrates the development metres associated with the LOM activities.

Figure 16.29 Life-of-Mine Development Schedule

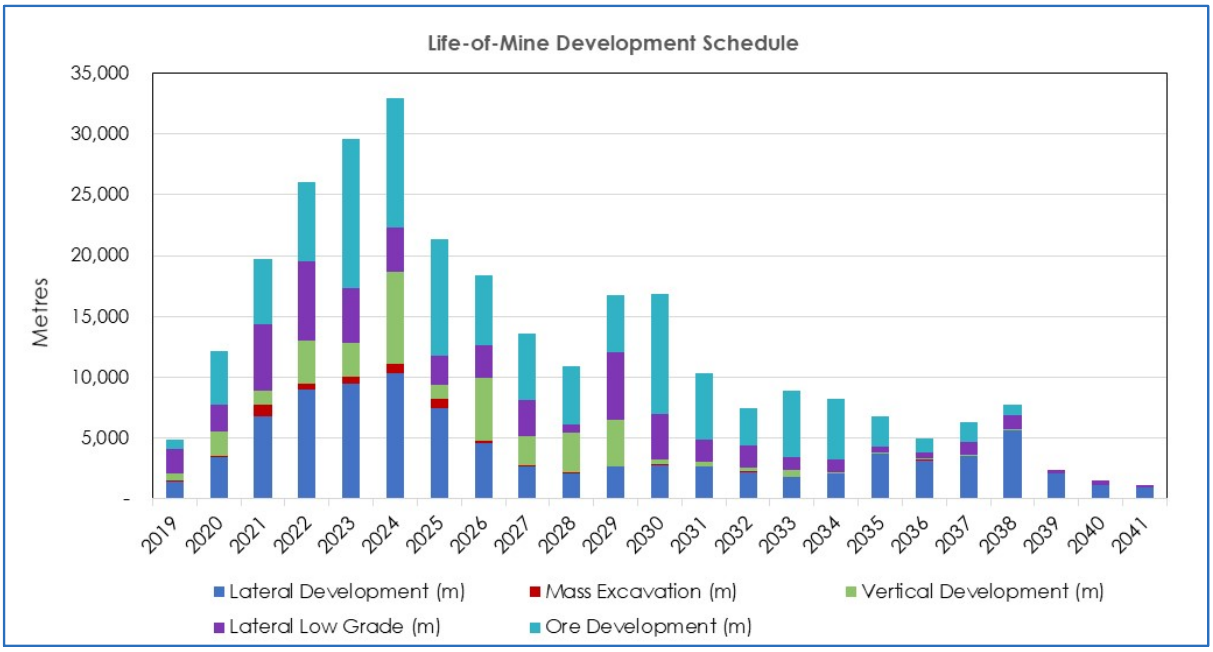


Figure by Stantec, 2020.

16.3.9 Mine Production Plan and Scheduling

Table 16.32 presents the annual preproduction and ramp-up targets and resulting scheduled tonnes that were set to meet the 6.0 Mtpa production rate.

Table 16.32 Preproduction and Ramp Up Targeted and Scheduled Tonnage

Production Schedule	Years	Targeted Tonnes (kt)	Scheduled Tonnes (kt)
Initial Production Mining (2019)	1	200	91.4
Ramp-Up (2020)	1	1,200	676.2
Ramp-Up (2021)	1	2,000	2,526.5
Ramp-Up (2022)	1	3,000	3,231.6
Ramp-Up (2023)	1	4,500	4,590.7
Initial and Ramp-Up Total	5	10,900	11,116.3

Figure 16.30 presents the resultant scheduled preproduction and ramp-up tonnes and TCu.

Figure 16.30 Preproduction and Ramp-up Tonnes and Copper Grade

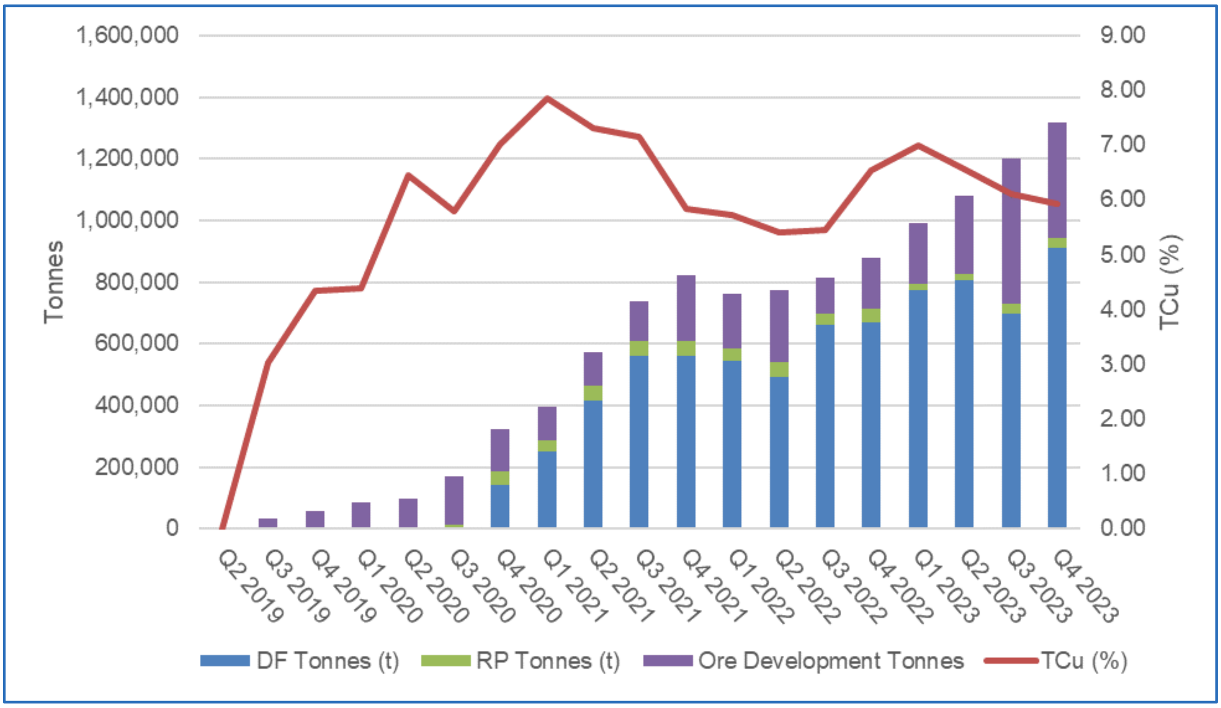


Figure by Stantec, 2020.

Life-of-Mine Production Schedule

Full production of 6.0 Mtpa is sustained for 15-years, starting in 2024 and tapering off after 2038 as the reserve is depleted. The mining blocks are scheduled so that a higher TCu value is achieved early in the mine life. Figure 16.31 illustrates the LOM production schedule and copper grade.

Figure 16.31 Life-of-Mine Production Schedule and Copper Grade

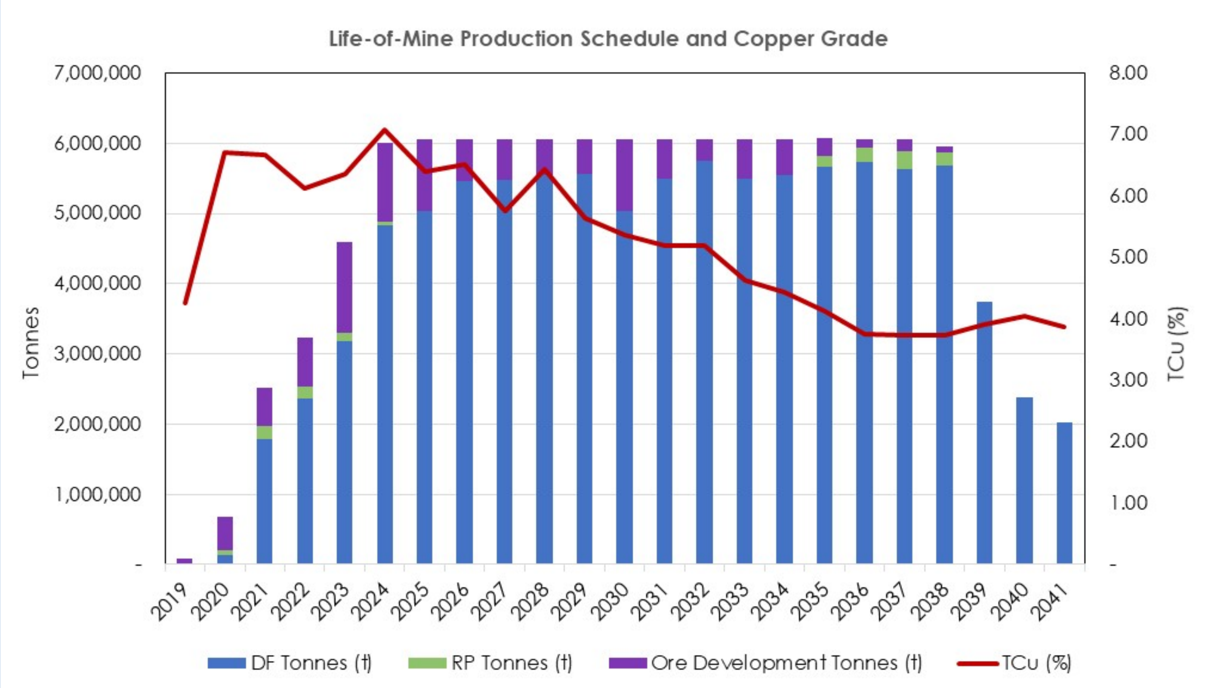


Figure by Stantec, 2020.

16.3.10 Underground Infrastructure

16.3.10.1 Mine Ventilation System

The underground mobile equipment fleet is diesel powered, and mine air cooling is required to maintain underground working air quality within the appropriate limits.

The following assumptions were considered in the ventilation design to maintain safe operating conditions underground and to abide by applicable legislative requirements. South African regulations for mine ventilation and industry best practices were considered, in the absence of DRC regulations.

- Primary ventilation system to be designed as a “pull” system. Main fans to be installed on surface and equipped with variable frequency drives.
- Airflow requirement for diesel engines will be provided with a minimum of 0.063 $m^3/s/kW$ airflow rate with utilisation factored in.
- Total airflow volume will include a 15% leakage factor and a maximum 15% contingency for when the mine is fully developed to maintain appropriate working temperatures and minimum velocities throughout all the openings.
- Airflow requirements will include allocations for fixed facilities to maintain minimum velocities and for dust controls, with crushers and rock breakers be provided with 24 m^3/s, each section of the conveyor belt 22 m^3/s and main workshops 47 m^3/s.

- Main workshops will be located to vent directly to exhaust air raise to minimise impact to operations in event of a fire.
- Auxiliary ventilation will use a forcing ventilation system with flexible or rigid ducting depending on duct length.
- Trucks and LHDs are assumed to have a heat load factor of 60%, while all other mobile equipment are assumed to have a heat load factor of 20%.
- Diesel engines are assumed to have a power conversion efficiency of 37%.

The mine layout schematic illustrating the location of the ventilation shafts is shown in Figure 16.32. The ventilation system is 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 minimising leakage.

Figure 16.32 Plan View of Ventilation Shaft Locations

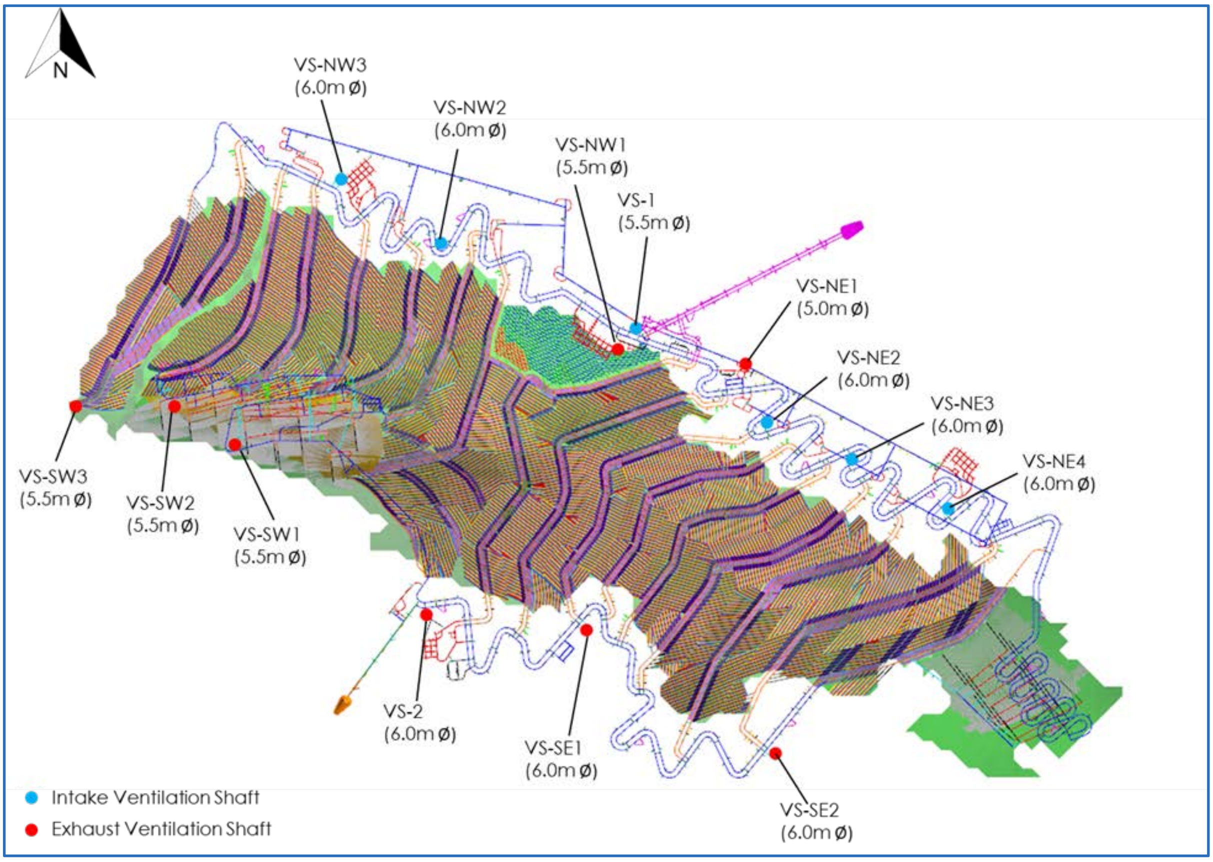


Figure by Stantec, 2020.

During pre-production period exhaust fans are installed on VS-1 until VS-2 is established. Once exhaust fans are installed on VS-2, the fans on VS-1 are removed and VS-1 is used as an intake raise.

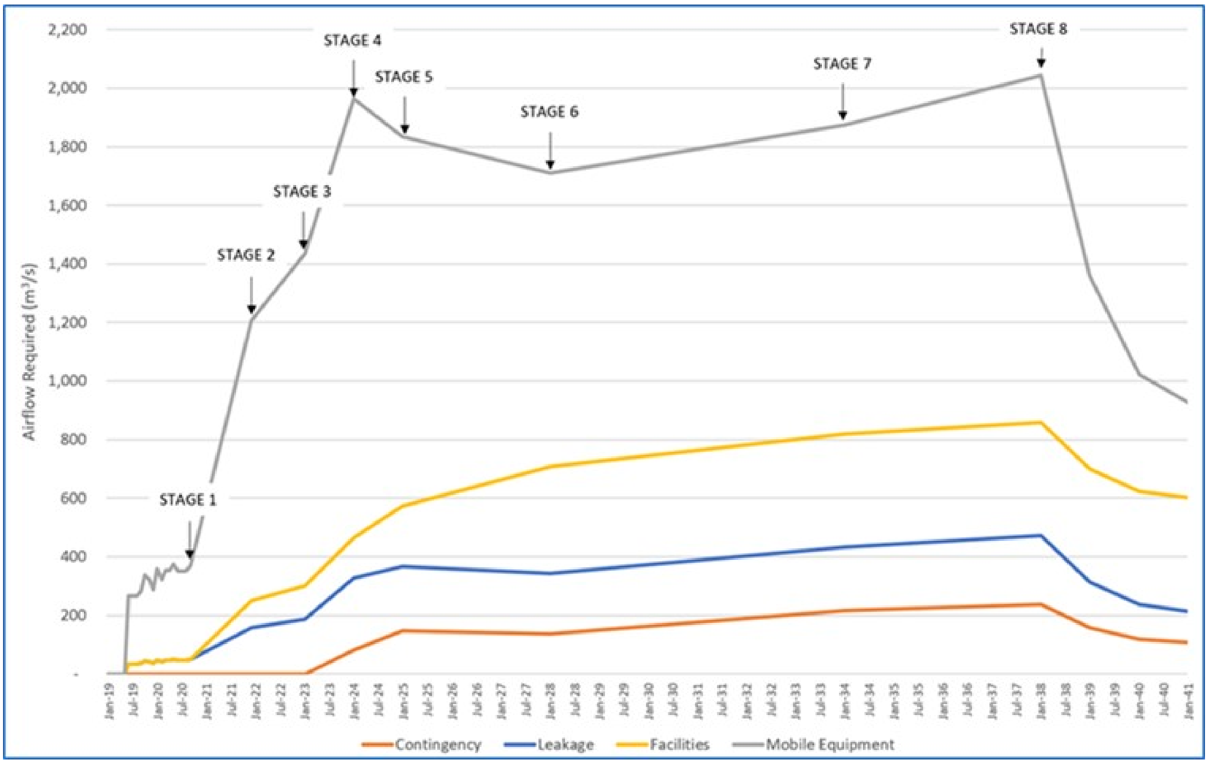
A summary of the primary ventilation fans is provided in Table 16.33. The LOM airflow requirements are shown in Figure 16.33.

Table 16.33 Primary Main Ventilation Fan Requirements

Raise Location	No. of Fans (in parallel)	Peak Airflow (m^3/s)	Peak Total Pressure at Collar) (Pa)	Estimated Power (kW)
VS-1	2	2 x 234	1,940	2 x 560
VS-2	3	3 x 234	3,000	3 x 890
VS-NW1	1	1 x 280	1,650	1 x 560
VS-NE1	1	1 x 270	1,750	1 x 560
VS-SE1	3	3 x 240	1,900	3 x 560
VS-SW1	2	2 x 260	2,500	2 x 890
VS-SE2	3	3 x 190	3,800	3 x 890
VS-SW2	2	2 x 260	2,500	2 x 890
VS-SW3	2	2 x 240	3,000	2 x 890

Note: Exhaust fans required on VS-1 during pre-production period only.

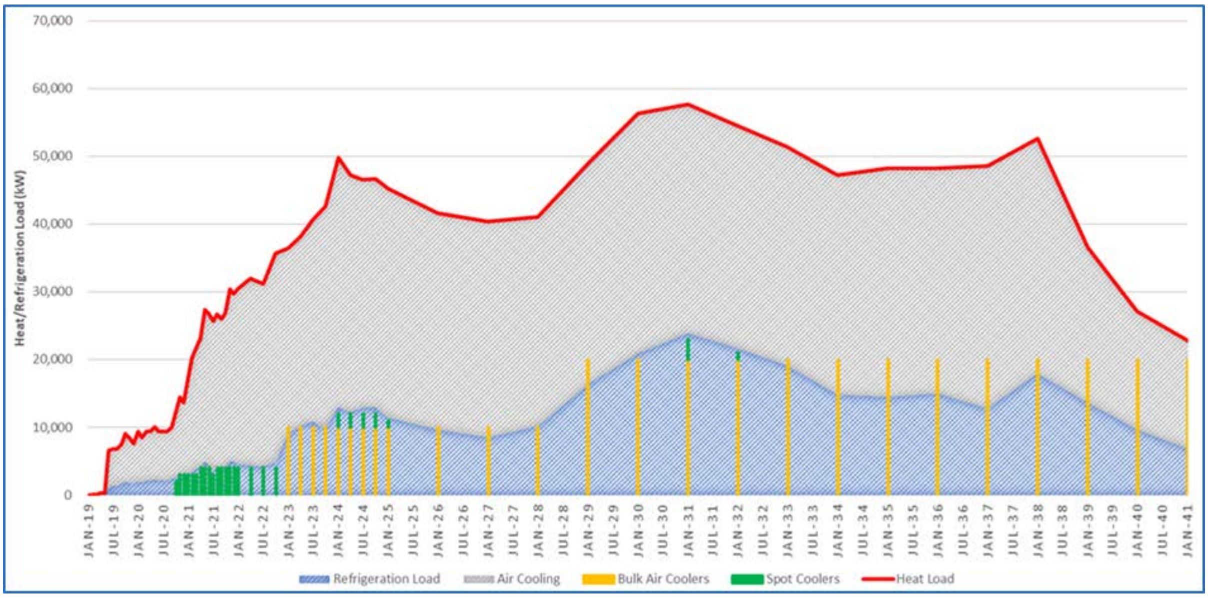
Figure 16.33 Life-of-mine Airflow Requirements



16.3.10.2 Cooling System Description

The heat load and refrigeration requirements of the mine are shown in Figure 16.34.

Figure 16.34 Life-of-mine Heat and Refrigeration Load



Bulk air and spot cooler requirements represented by column at each modelled period.

The bulk air cooling will be supplied from two refrigeration plants located west and east, each with a nominal air-cooling duty of 10 MWR (megawatts of refrigeration). This will provide 20 MWR total installed nominal refrigeration. Each refrigeration plant will comprise of two BACs with a nominal air-cooling duty of 5 MWR. The western plant will be installed first and the eastern plant will be phased in as the mine develops.

Auxiliary Ventilation

Auxiliary fans with ducted ventilation tube will be used to provide ventilation to the working areas. The estimated fan requirements for auxiliary ventilation to support development and production are provided in Table 16.34.

Table 16.34 Auxiliary Ventilation Fan Requirements

Location	Fan Qty	Flow per Fan (m^3/s)	Fan Total Pressure (Pa)	Fan Size diameter (m)	Duct Type	Estimated Power (kW)
Development Headings	26	36	2,300	1.40	Flexible	110
Room-and-pillar headings	8	10	4,300	0.76	N/A (Jet Fan)	45
Drift-and-fill Headings	56	53	1,050	1.60	Rigid and Flexible	75

16.3.10.3 Underground Dewatering

The underground dewatering system consists of a series of collection dams and cascade transfer dams on the north and south perimeter drifts. These will pump to underground and surface settler systems.

Mechanical de-gritting only will occur at the pump stations with installed multistage pumps and dirty water pumping. The dirty water will be cleaned on surface by thickeners or settlers located near the portal at the existing settling dams.

An overview of the underground dewatering system is illustrated in Figure 16.35.

Figure 16.35 Underground Dewatering System

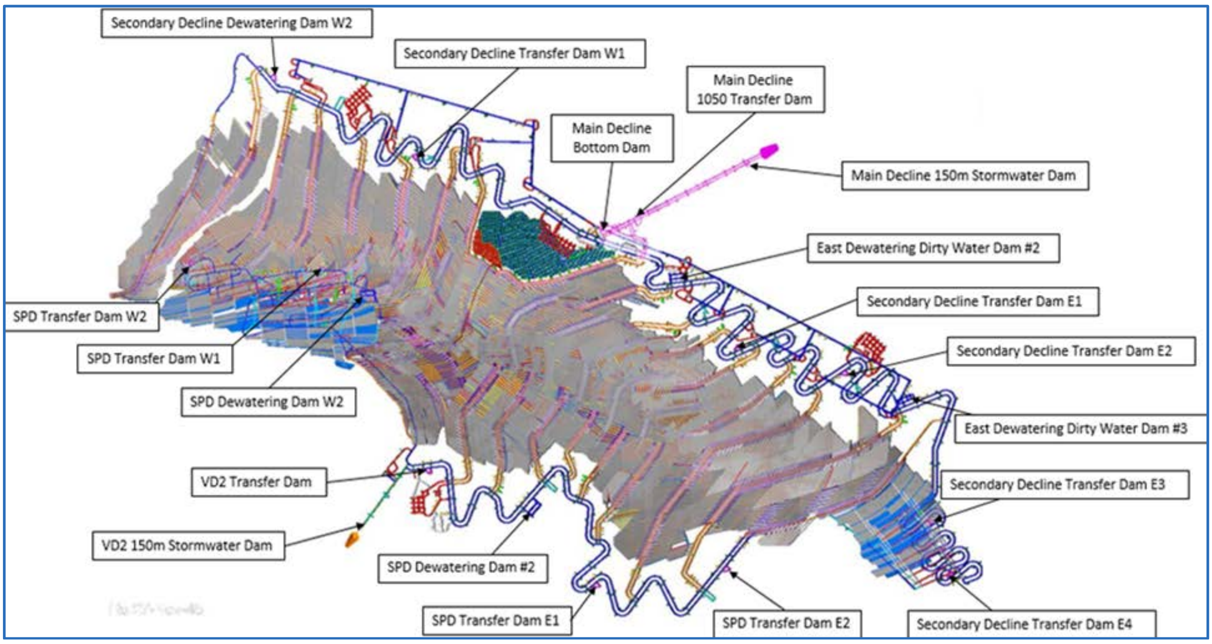


Figure by DRA, 2020.

Inflow Rates and Basis of Designs

Groundwater is estimated by Golder Associates Africa (Pty) Ltd to reach peak inflows of 660 L/s. Additional water sources are expected to be from: drill water (26 L/s), Washdown and dust suppression (5 L/s), backfill flush water and seepage (9 L/s).

Water in ventilation and water in ore is expected to each account for 5% of water removed from the mine.

Modelling inflow and pumping rates indicates a transfer dam capacity of 3,000 m^3 with an average pumping time of 18 h/day is sufficient for dewatering.

Production and development dewatering

The dirty water at development faces or in production areas is collected by submersible pumps. The submersible pumps (7.5 kW) move the water to a 2 m^3 portable pump skid equipped with vertical spindle pumps (37 kW). Larger 8 m^3 pump skid equipped with 132 kW pumps, collect water from two of these smaller skid pumping systems and pump into nearby transfer dam or dewatering dam.

Transfer Dams

Transfer dams are positioned on the perimeter drifts and are used as cascading and collection dams. These dams collect water from development and production areas and transfer the water to the next transfer dam or into a dewatering dam.

Transfer dams are equipped with three pump trains with space provided for a fourth installation. Each pump train comprises three, single-stage, dirty water centrifugal pumps placed in series which are capable of delivering between 120 L/s and 170 L/s at a total head of between 148 m and 217 m (vertical head between 130–177 m).

The transfer dams are also 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.

Mine Dewatering Dams – Centrifugal Pumps

Mine dewatering dams pump directly to surface via boreholes, and overland pipes transfer the water to the surface settling dams or surface settler. Dewatering dams are equipped with a buffer dam where dirty water collects before it is fed into one of the two collection dams. No de-gritting walls are constructed at the inlet to these dams as the dewatering pumps are equipped to handle dirty water. An agitation system will keep fines in suspension in the main collection dam until the water is pumped out. These dams are constructed in two separate sections to allow grit to be cleaned out using an LHD.

The mine dewatering dams will be equipped with three pump trains and space are provided for a fourth installation. Each pump train will comprise three, single-stage, dirty water centrifugal pumps placed in series. Pumps selected for these pump stations will be of a different design than those in the transfer pump stations due to an elevated head. Each dewatering dam pump train will be capable of delivering 200 L/s at a total head of between 371–406 m (vertical head between 326–346 m).

Mine Dewatering Dams – Multistage Pumps

Secondary Decline Dewatering Dam W2 and the East Dewatering Dirty Water Dam No. 3 have total heads of 716 m and 828 m respectively. These dewatering dams are equipped with multi-stage pumps.

These dams are equipped with de-gritters to prevent larger particles from entering the dam.

A summary of the transfer and dewatering dams is provided in Table 16.35.

Table 16.35 Underground Dewatering Dams

Dam	Pump Station Type	Inflow (L/s)	Maximum Capacity (L/s)	Pump to
North-east				
Main Decline 1050 Transfer Dam	Centrifugal	349	800	Surface
East Dewatering Dirty Water Dam No. 2	Centrifugal	164	600	Surface
Secondary Decline Transfer Dam E1	Transfer dam	132	360	East Dewatering Dirty Water Dam No. 2
Secondary Decline Transfer Dam E2	Transfer dam	68	510	Secondary Decline Transfer Dam E1.
East Dewatering Dirty Water Dam No. 3	Multistage	199	720	Surface
Secondary Decline Transfer Dam E3	Transfer dam	136	360	East Dewatering Dirty Water Dam No. 3.
Secondary Decline Transfer Dam E4	Transfer dam	72	510	Secondary Decline Transfer Dam E3.
South-east				
VD-2 Dewatering Dam	Centrifugal	151	600	Surface
SPD Dewatering Dam No. 2	Centrifugal	187	600	Pump to surface.
SPD Transfer Dam E1	Transfer dam	139	360	SPD Dewatering Dam No. 2.
SPD Transfer Dam E2	Transfer dam	64	510	SPD Transfer Dam E1.
South-west				
SPD Dewatering Dam W2	Centrifugal	157	600	Surface
SPD Transfer Dam W1	Transfer dam	119	450	SPD Dewatering Dam W2.
SPD Transfer Dam W2	Transfer dam	73	360	SPD Transfer Dam W1.
North-west				
Secondary Decline Transfer Dam W1	Transfer dam	95	360	Main Decline 1050 Transfer Dam.
Secondary Decline Dewatering Dam W2	Multistage	211	720	Surface

16.3.10.4 Rock Handling System

The underground rock handling system consists of northern and southern infrastructure. In the northern infrastructure, ore is delivered to the decline conveyor systems by a combination of LHDs and underground dump trucks. The northern infrastructure is comprised of three component conveyor systems, the western rock handing, eastern rock handing, and main decline rock handing.

The southern infrastructure comprises the southern rock handling systems and VD-2 rock handling.

The southern rock handling system is located in the south-western part of the mine and comprises LHD tips points into ore passes which are equipped with truck load-out points below. Trucks load and haul from the loadout points up the southern decline, (the VD-2 rock handling system). The VD-2 rock handling system is a truck haulage system but has been designed with the capabilities to install a conveying system.

The main components of rock handling are illustrated in Figure 16.36.

Figure 16.36 Underground Rock Handling System Overview

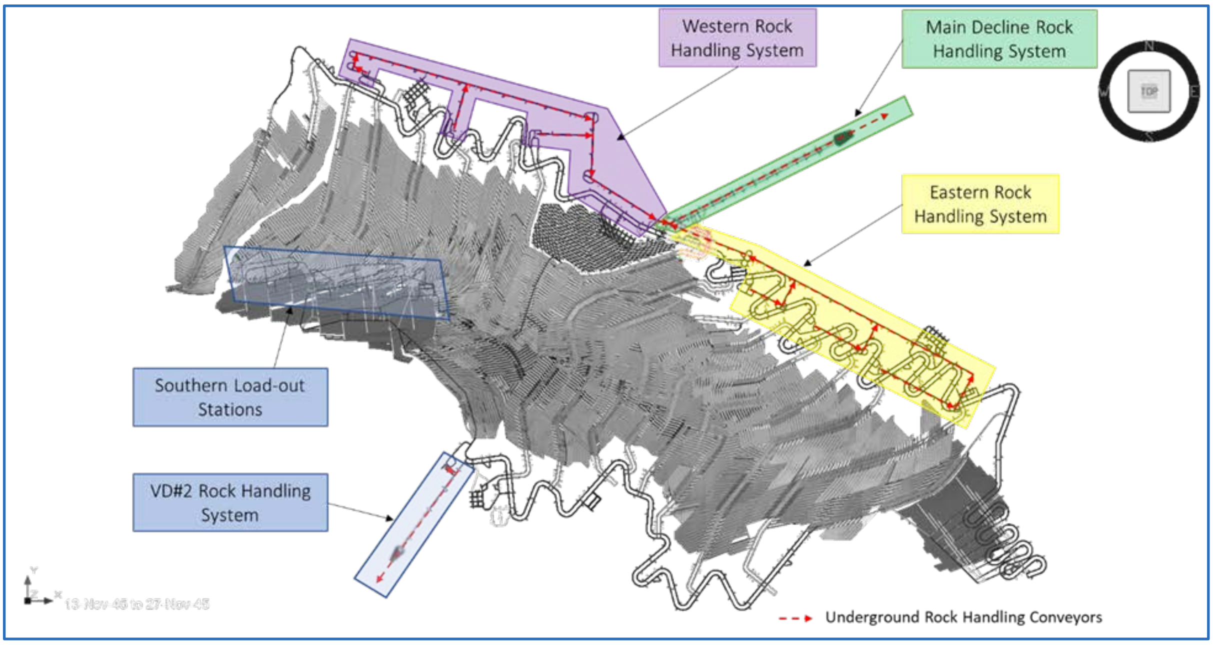


Figure by DRA, 2020.

Western Rock Handling

The western rock handling system comprises three truck tips that feed rock through a series of conveyor belts into the Main Decline West Transfer Bin. A static scalping grizzly, fixed hydraulic-boom rock breaker, apron feeder, sacrificial conveyor, and tramp metal magnet is located at each truck tip.

The western rock handling system is designed to deliver a total capacity of 2,000 t/h.

Eastern Rock Handling

The eastern rock handling system comprises three truck tips that will feed ore through a series of conveyor belts and an east jaw crusher system into the Main Decline East Transfer Bin. The eastern system also has a static scalping grizzly, fixed hydraulic-boom rock breaker, apron feeder, sacrificial conveyor, and a tramp metal magnet is located at each truck tip.

The eastern rock handling system is designed to deliver a total capacity of 2,000 t/h

Main Decline Rock Handling

The main decline rock handling system receives feed from the east and west transfer bin feed conveyors as well as from two truck tips. Rock received in the two transfer bins is controlled by apron feeders and fed through one common sacrificial conveyor onto the main decline conveyor belt which transports 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.

Southern Rock Handling

The southern rock handling infrastructure comprises southern load-out stations and the VD-2 Decline rock handling system. The south-western rock handling system comprises LHD dumps and truck loadout stations.

Dump trucks transport the broken rock from the load-out stations to surface through VD-2 Decline. The opportunity exists to install a conveyor system in the VD-2 Decline.

Mining Surface Rock Handling System

The mining surface rock handling system receives ore from underground via the main decline conveyor.

A series of conveyor belts transport the rock directly or indirectly to the ROM stockpile, which serve as the main buffer feed to the processing plant.

A bypass stockpile is located on surface equipped with a reversable conveyor to separate material based on grade. These stockpiles will be loaded out and stored at the bulk stockpile for later reclamation.

The surface rock handling system is designed for a 2,000 t/h capacity.

Truck Tips

Underground truck tips are equipped with a 350 mm x 350 mm static scalping-type grizzly screen. Screened material will pass through the grizzly into the transfer bin and onto an apron feeder. Screened-out oversize rocks will be broken into the appropriate size using a fixed hydraulic-boom rock breaker. Tramp metal that remains on the static grizzly will be removed using a fixed hydraulic-boom magnet and deposited into a tramp metal cassette for removal.

The truck tips apron feeders feed onto 1.8 m W sacrificial conveyors.

Ore Passes and Transfer Bins

There are two ore passes located on the southwestern area of the mine. These two ore passes are equipped with a combination of five LHD tipping areas through 350 mm x 350 mm static scalping grizzly screens. Load-out stations for loading 60 t underground articulated dump trucks are located at the bottom of each ore pass. These load-out stations are equipped and operated with radial doors to control the feed rate from the ore pass into the dump trucks.

There are also ore passes located at each one of the nine truck tips. These have limited capacity referred to on the project as transfer bins.

Crusher

A crusher will be located on the eastern rock handling system and receives all rock feed from the respective eastern production sections. A Jaw crusher has been selected since they are easy to maintain and more reliable than other viable options.

A vibrating grizzly feeder with a 220 mm aperture removes undersize prior to the crusher feed and will prevent overloading and excessive wear to the crusher liner plates. The 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.

Crusher chamber throughput rates are expected to be less than 250 t/h through the crusher and 1,750 t/h as undersize bypass, totalling the required 2,000 t/h through the rock handling system.

16.3.10.5 Compressed Air System

The compressed air system will not include a mine-wide reticulation system from a main surface facility. Every workshop and satellite workshop will have its own stationary compressor. Stationary electric compressors have been sized for specific application requirements for different workshops. The electric compressors will be equipped with a receiver and ancillary piping.

Processes like valve actuation, which typically use compressed air, will be substituted with electrical actuators or hydraulic actuators. Production mobile machines and all other equipment will be equipped with on-board compressors. Portable refuge stations will be self-contained, with no main line compressed air system connection.

16.3.10.6 Water Systems

Potable Water

A surface potable water treatment plant and storage tank feed a 100 mm galvanised steel pipe that transports potable water underground via the main decline. The potable water line provides clean drinking water as well as gland seal water to the underground pumping systems.

Service Water

Service water is obtained from the existing surface mine service water tank, which gravitates underground via two 200 mm galvanised steel pipes. Service water consumption is estimated based on 33 crews drilling using 3 L/s, for a total consumption of 60 L/s over the Life-of-mine.

Fire Protection System

The following two fire-water ring mains are provided 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 supplies 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, and hydraulic power packs.

Protection of all other areas underground is achieved via portable handheld equipment with particular reference to the following areas:

- Workshops.
- Mini-substations.
- Underground motor control centre (MCC).
- Refuelling bays.

The equipment fleet have built-in extinguisher systems along with hand-held units.

Substations on surface and underground are protected by means of fire suppression systems utilising heat and smoke detection combined with an automated extinguishing system.

16.3.10.7 Materials Handling Logistics

Delivery vehicles report to the main gate and are then directed to the correct offloading area. Most materials are delivered directly to one central main store on surface. Materials are distributed underground by being loaded into cassettes according to area and section.

Certain consumables such as maintenance consumables are stored at a store in the workshop area. Mining consumables such as roof bolts, resin, drill steels, are stored in mining storage areas.

16.3.10.8 Workshops

Mobile equipment maintenance is supported by two surface workshops, four underground workshops, and temporary satellite workshops located underground near the working face. Equipment wash bays are located at the entrance of each workshop.

The surface workshops are a heavy vehicle workshop and a light vehicle workshop.

There are two centrally located underground workshops on the perimeter of the orebody with one on the northern perimeter (UG Workshop – Central North) and one on the southern perimeter (UG Workshop – Central South). The remaining two underground workshops are located on the north-east and north-west perimeter drifts of the orebody (UG Workshop – East and UG Workshop – West).

As the extents of the mine increases, there is a need for a limited number of satellite workshops equipped to perform daily maintenance on slow-moving mobile equipment such as jumbos and bolters. The satellite workshops have limited capacity with a single-bay. Concrete floors, lighting, and a hoisting arrangement is incorporated in their design.

Field service and repair, fuel and lube trucks provide minor repair and lubrication services at the working face or other points of use.

16.3.10.9 Explosives Magazine

Surface delivery of Class 1 explosives to the surface explosives magazine is via purpose-built explosives cassettes. Once the pallets are offloaded on surface, they are reloaded for transport underground.

The offloading facilities on surface are 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 are positioned at the magazine.

There are two magazines located underground, each with three bays, one for detonators, one for package explosives and one bulk emulsion.

Until the development of the appropriate underground magazine, the emulsion and sensitiser will be stored in tanks on surface located near the portal. Emulsion trucks can deliver emulsion every second day; however, sufficient storage capacity for four days is provided.

AEL Mining Services provided the design and costs for a suitable emulsion 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 dropped in separate HDPE pipes through the same borehole.
- Emulsion cassettes are used to store explosives near working areas or in the emulsion storage magazine.
- Emulsion and sensitiser transported using utility vehicles to underground working areas.

The proposed emulsion delivery system is well understood and in use at various other operations.

16.3.10.10 Refuge Stations and Emergency Egress

Refuge stations are required to house underground mining personnel in a secure, hazard-free location during emergency conditions. Refuge stations are provided underground in all working areas and travel ways between working areas. No person underground is further than 750 m from the nearest refuge station in the event of an emergency. Two types of refuge stations are used: permanent refuge stations in the underground workshops, and portable refuge stations for all underground areas away from the underground workshops.

Portable refuge stations are 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, carbon monoxide scrubbing, cooling, and gas monitoring.

Portable refuge stations will be relocated to maintain compliance as the mining development faces advance.

The mine issues an individual self-contained self-rescuer (SCSR) to each employee working underground and trains them in their use.

General emergency escape plans follow South African and United States 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 followed:

- The two separate and independent shafts or outlets to surface required in terms of regulation 6.1.1.
- Shall not at any point be nearer to each other than 9.2 m.
- Shall be provided with proper arrangements, which will be kept constantly available for use, to enable persons to travel to and from the surface.
- 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 are part of the emergency escape strategy. Simulated emergency exercises (training) is conducted at the mine at regular intervals.

16.3.10.11 Toilet System

Portable toilets are located in strategic locations underground. A purpose-built toilet facility is connected to each permanent refuge station at the underground workshops and sealable to the outside environment in the event of emergency. The toilets are serviced by a mobile effluent removal cassette and transported to surface for discharge into the waste system by a mobile cassette carrier.

16.3.10.12 Electrical Substations and Power Distribution

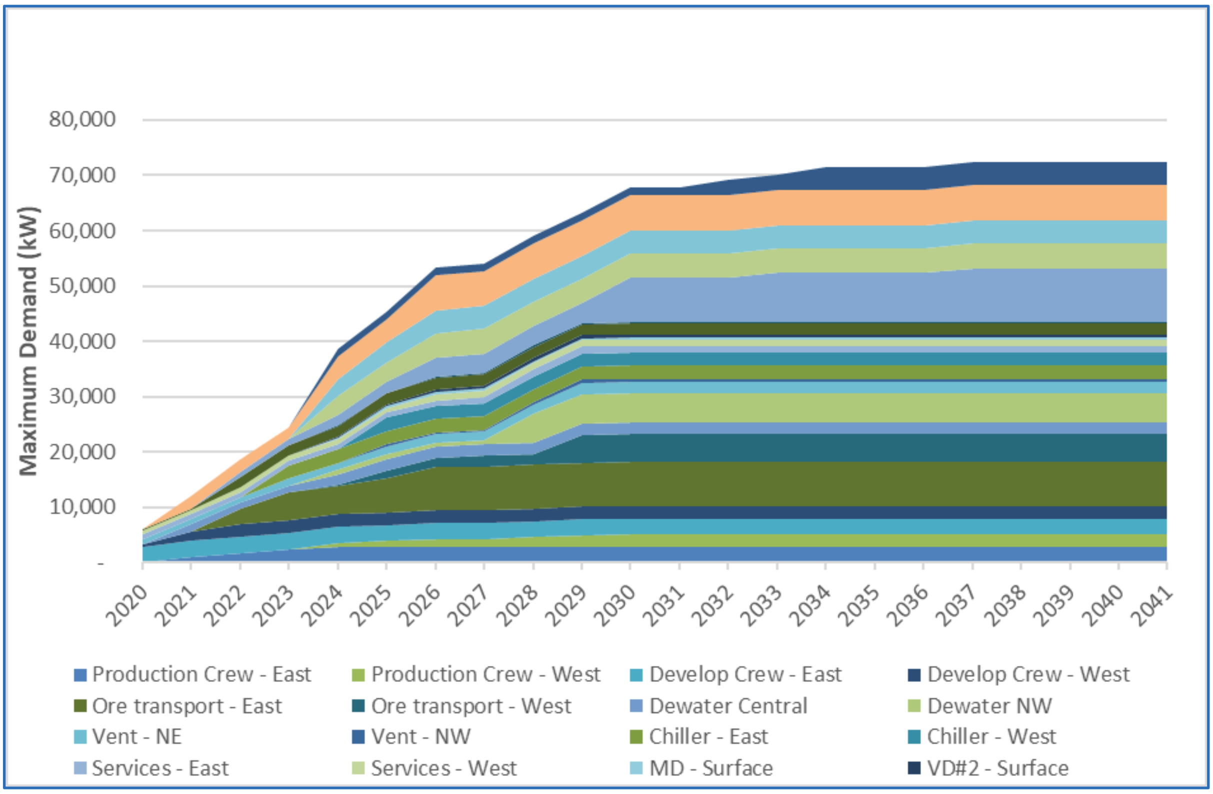
The mine electricity is supplied via two substations. Substation 1 was installed during Phase 1 development and supplies developmental and permanent power. Substation 2 will supply the mine air cooling plants and backfill paste plants and will provide 11 kV power distribution.

Power is distributed at the following voltages:

- 220 kV – SNEL supply voltage.
- 110 kV – SNEL supply voltage from the mobile transformer.
- 33 kV – Surface distribution.
- 11 kV – Medium voltage distribution underground and surface.
- Secondary distribution to mining equipment – 1,000 V.
- Secondary distribution to surface and underground infrastructure equipment – 690 V.
- Low voltage distribution – 400/230 V.

The maximum power demand for the mine is 72.4 MW. The power demand profile is shown in Figure 16.37.

Figure 16.37 Maximum Power Demand for Mining



16.3.10.13 Emergency Generator Plant

Emergency power is supplied by 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 (3.8 MW).

Surface loads consist of surface production fans, backfill paste plants, booster stations, and the cooling plants for the underground mine.

16.3.10.14 Communications and Control / Automation Systems

The backbone for the communications system is based on a redundant fibre network. This system supports all voice and data communication requirements for the mine. Radio communication for the mine are provided over a leaky feeder system, which is distributed throughout the entire mine for communication purposes, incorporating hand-held and fixed radios. The leaky feeder also supports systems such as proximity detection, vehicle detection, and ventilation-monitoring.

The mining control room is located on surface for monitoring and 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 are centrally located in the engineering rooms on surface with the exception of the main surface ventilation fans which are provided with a detailed PLC located within the substation at each vent shaft.

16.3.10.15 Underground Access Control

Upon entering the mine site through the surface access-controlled complex, mining personnel proceed to the change house and lamp room where they tag-in/tag-out. Miners' locations are monitored in the control room by an electronic tracking system that is integrated into the cap lamps.

16.3.11 Equipment

All equipment is sized for a 6.0 Mtpa case to support room-and-pillar and drift-and-fill mining methods. All ore material is conveyed out of the mine via a series of truck tips, ore passes, and conveyor belts. Waste material is transported using mobile truck haulage equipment.

Criteria considered in equipment selection includes 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 feasibility level of accuracy and cover the major components for the operation.

16.3.11.1 Mobile Equipment

The mobile equipment is diesel-powered, rubber tired. Typical development equipment such as jumbo drills is used for the drilling and ground support. Explosives trucks transport explosives and detonators to the headings. LHDs load the blasted material and transport it to a re-muck stockpile or the truck tips. LHDs re-handle material transported to re-muck stockpile into trucks where the material is transported to truck tips or a designated area, depending on whether the rock is ore or waste.

In areas where the required development drift centreline height is 4.5 m H or less, low-profile equipment is used. Drifts that have 4.5 m or greater centreline heights such as the first lift in the drift-and-fill areas and all primary development 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 16.36.

Table 16.36 Mobile Equipment List

Description	Maximum Number Required	Number of Units to Replace	Number of Rebuilds
Development and Production Equipment			
Double-Boom Drill Rig – Standard Profile1	33	26	N/A
Double-Boom Drill Rig – Low Profile	11	5	N/A
LHD – 21 t	29	20	44
LHD – 14 t	8	3	5
Haul Truck – 63 t	11	39	47
Concrete / Shotcrete Mixer Truck	10	0	0
Shotcrete Sprayer	10	0	0
Scissor Lift	15	10	11
Charmec – Explosives Loading Truck	10	17	N/A
Mine Support Equipment			
Explosives Transport Truck	10	18	N/A
Agicar – Concrete Mixing Truck – 12 m^3	4	7	N/A
Shotcrete – Backfill	4	7	N/A
Scissor Lift – Backfill	10	21	N/A
LDVs	55	213	N/A
Personnel Carriers	13	33	N/A
Grader	3	5	N/A
Utility Equipment – Material	9	25	N/A
Utility Equipment – Maintenance	7	18	N/A
Telehandlers	10	26	N/A
Underground Mobile Crane	3	3	N/A
Skidsteer	7	18	N/A

16.3.11.2 Fixed Equipment

Table 16.37 lists the main fixed equipment that will support the mining operation at full production.

Table 16.37 Fixed Equipment

Services	Description
Materials Handling	Truck Tips
	Static Hydraulic Rock Breakers
	Static Hydraulic Tramp Iron Magnets
	Apron and Vibrating Feeders
	Conveyors
	Self-Cleaning Belt Magnets
Ventilation	Main Fans
	Development and Production Fan
	Mine-Air Cooling Facilities
Mine Service Water	Centrifugal Pumps
Mine Dewatering	Vertical Spindle (Sump) Pumps
	Skid-Mounted Pumps and Tanks
	Centrifugal Pumps
	Multistage Clearwater Pumps
Electrical and Communications	Main Substation
	Motor Control and Mine Power Centres
	Leaky Feeder System
Safety and Miscellaneous	UG Safety Equipment
	Portable Refuge Stations
Surface Facilities	Fuel and Lubrication Facility Equipment
	Concrete and Shotcrete Facility Equipment
	Temporary Emulsion Storage Facility Equipment
	Permanent Emulsion Storage Facility Equipment
	Surface Heavy Vehicle Workshop Equipment
	Surface Light Vehicle Workshop Equipment
Underground Facilities	UG Workshop – Central North Equipment
	UG Workshop – Central South Equipment
	UG Workshop – East Equipment
	UG Workshop – West Equipment
	Satellite Shop Jib Cranes/Fire Doors
	Main Emulsion Storage Facility Equipment
	South Emulsion Storage Facility Equipment
	Concrete/Shotcrete Facility Equipment
	Fuel and Lubrication Facility Equipment

16.3.12 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 for example 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 engineering, procurement and construction management (EPCM) team.

Figure 16.38 illustrates the average annual personnel requirements for the mining over the Life-of-mine.

Figure 16.38 Contractor Versus Owner Personnel Summary

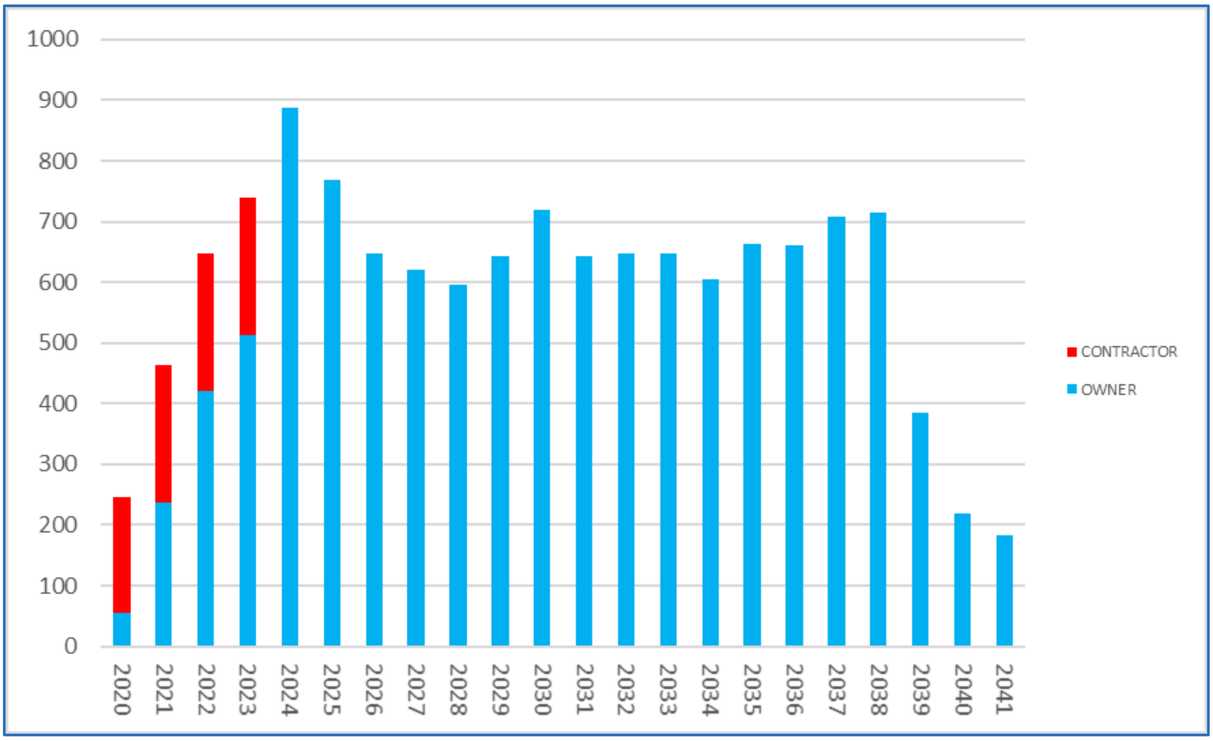


Figure by Stantec, 2020.

Direct and indirect labour 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 two shifts/day and 360 day/year for both Contractor and Owner crews.

It is Stantec's understanding that workforce training is an extremely important requirement for the success of the mine. The feasibility study assumes a trained workforce that requires only periodic refresher training.

Personnel are presented as direct and indirect staffing for both Contractor and Owner teams. Staffing is further classified as expatriate hourly, expatriate staff, local hourly, and local staff.

The Owner's Project Team oversee the work performed by the Contractor and coordinated by the EPCM Contractor. This includes labour, daily expenditures, and all equipment operating costs. Costs for the Owner's Team, labour, and associated non-labour costs are included in indirect costs. All production activities will be performed by Owner personnel.

During the preproduction period, Contractors complete all construction and include up to six crews used for waste development activities. Contractor involvement in waste drift development will be complete at the end of Q4'23.

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: Represents a "miner's pool" that is required on site to cover hourly labour during times when individuals are on vacation, sick, absent, or in training. This amounts to approximately 15% of the annual hourly personnel requirements.

16.4 Kansoko Underground Mining

The main mining methods at the Kansoko Mine 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 and trackless mining operation. Access to the mine is planned to be via a twin decline system from Kansoko Sud portal.

Kansoko portion of the Kakula-Kansoko 2020 PFS production schedule was developed based upon a combined 7.6 Mtpa production rate and reducing to 6.0 Mtpa production rate when Kakula is mined out. This plan focused on a high-grade scenario.

The Kansoko development schedule initially is 1.6 Mtpa creating a combined 7.6 Mtpa for the Kakula-Kansoko 2020 PFS. The Kansoko schedule then focuses on establishment of necessary mine services and support infrastructure to set up the initial production mining areas and ramp-up to 6.0 Mtpa ore production and associated development waste.

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.

16.4.1 Mining Methods

The Kansoko Mine orebody geometry indicates different orebody thicknesses and slopes. The orebody dips between 0–35°, with an average dip of 17°. The thickness varies between 3.0–6.0 m.

Access to the mine is via a twin declines system from the Kansoko Sud Portal, to support mining the Kansoko Mine deposit. Besides primary development, the two mining methods for this orebody are room-and-pillar mining and controlled convergence room-and-pillar mining.

16.4.1.1 Room-and-Pillar

Room-and-pillar mining is 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 runs 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 are 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, remain accessible if maintained and ventilated.

The extraction ratios for room-and-pillar mining was 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 W rooms. A mining height of 6 m has been assumed. Eight rooms are required to meet the maximum panel span of 152 m, which is bounded by 20 m regional pillars. A maximum strike length of 504 m has been allowed. A row of regional pillars 15 m wide is required before the next panel is started.

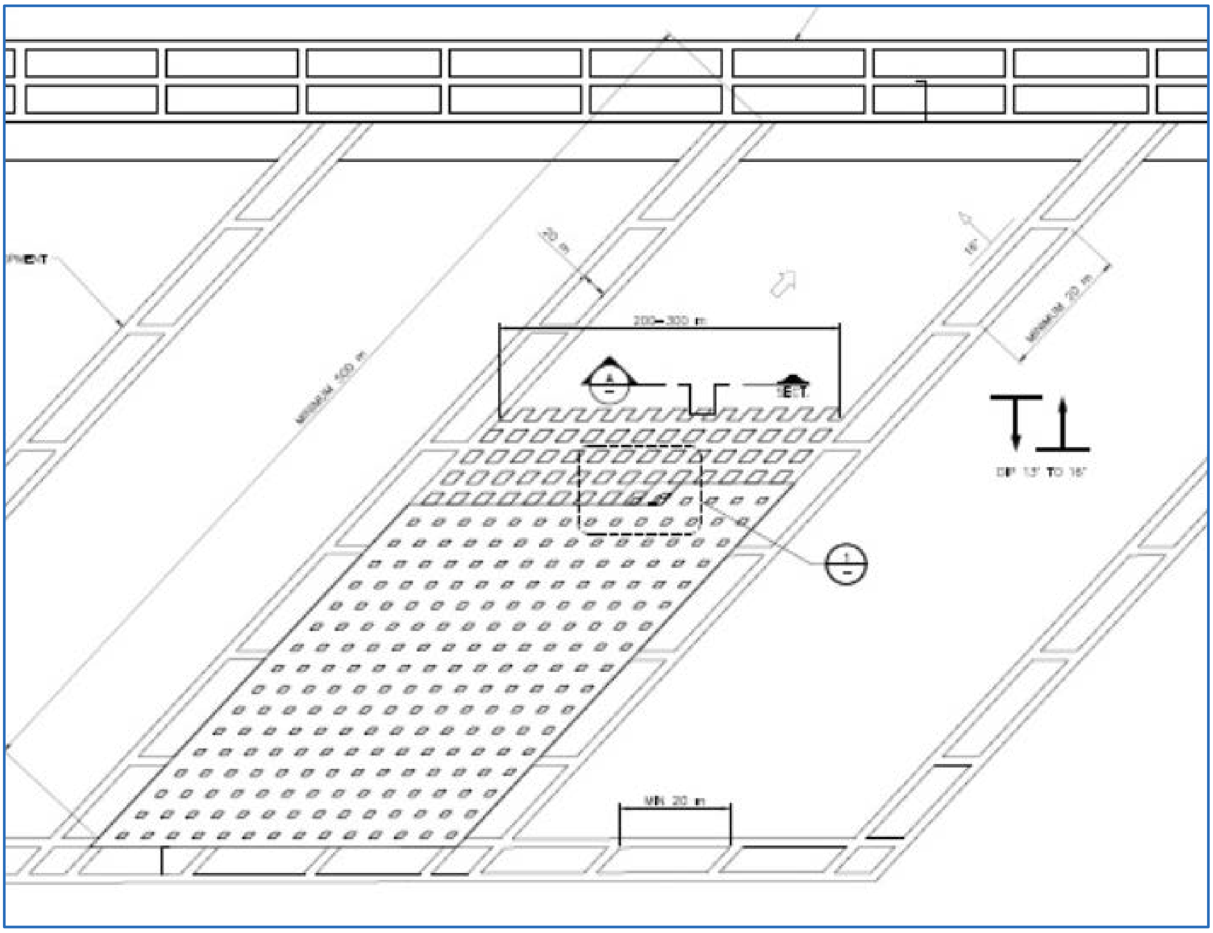
The calculated extraction ratios for areas mined using the room-and-pillar method vary with depth and range from 53–70%.

16.4.1.2 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 W and a minimum of 500 m L, 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 16.39 is a typical controlled convergence room-and-pillar panel, with a mining direction advancing toward the access.

Figure 16.39 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 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.

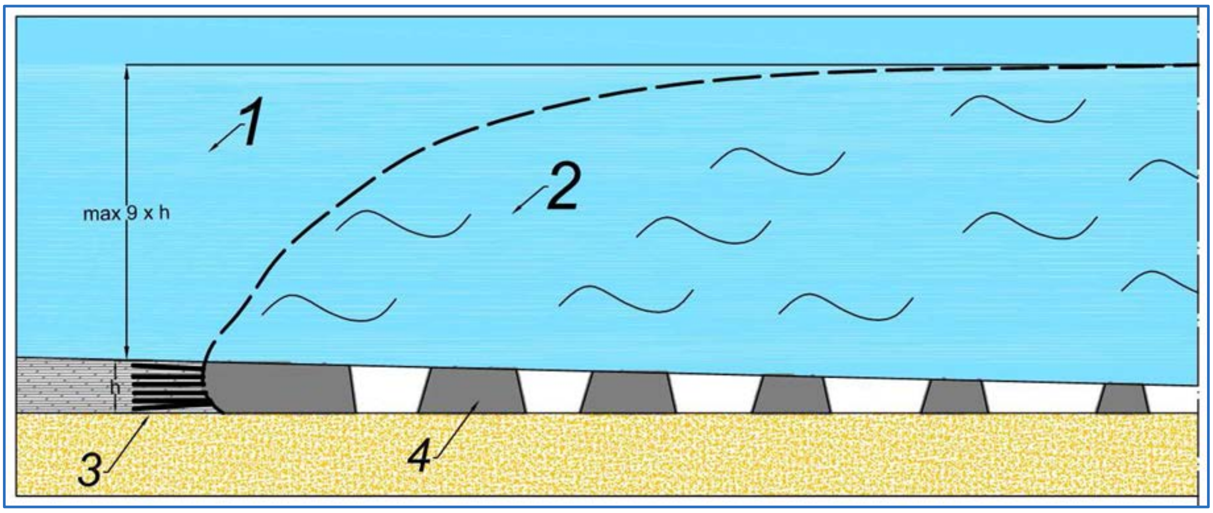
These pillars are reduced through scraping or drill and blast to a remnant pillar size. The remnant size is the minimum size pillar that is allowed to be scraped while still ensuring the overall front is supported and continues to converge. Once remnant size pillars are established, personnel and equipment are prevented from working in the 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) (Figure 16.40). 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 was based on the dip and the height of the panels and Cuprum's resulting pillar design.

Figure 16.40 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.

16.4.2 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 hanging wall dilution shell on top of the production panel shape and 2 x 1.0 m footwall 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 W x 6.0 m H and a conveyor drift with cross-sectional dimensions of 7.0 m W x 6.0 m H. 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 hanging wall 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 H, the angle of the back and the floor are adjusted. The result is a slightly increased footwall dilution and slightly reduced hanging wall dilution. This occurs for all 3.0 m high thicknesses with a dip greater than 12° and all 3.5 m H 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 hanging wall 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 hanging wall 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 hanging wall. 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.

16.4.2.1 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.

16.4.3 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°, which is 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 W x 6.0 m H 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 16.41 shows the portal, declines, and underground infrastructure.

Figure 16.41 Kansoko Mine Underground Access Infrastructure

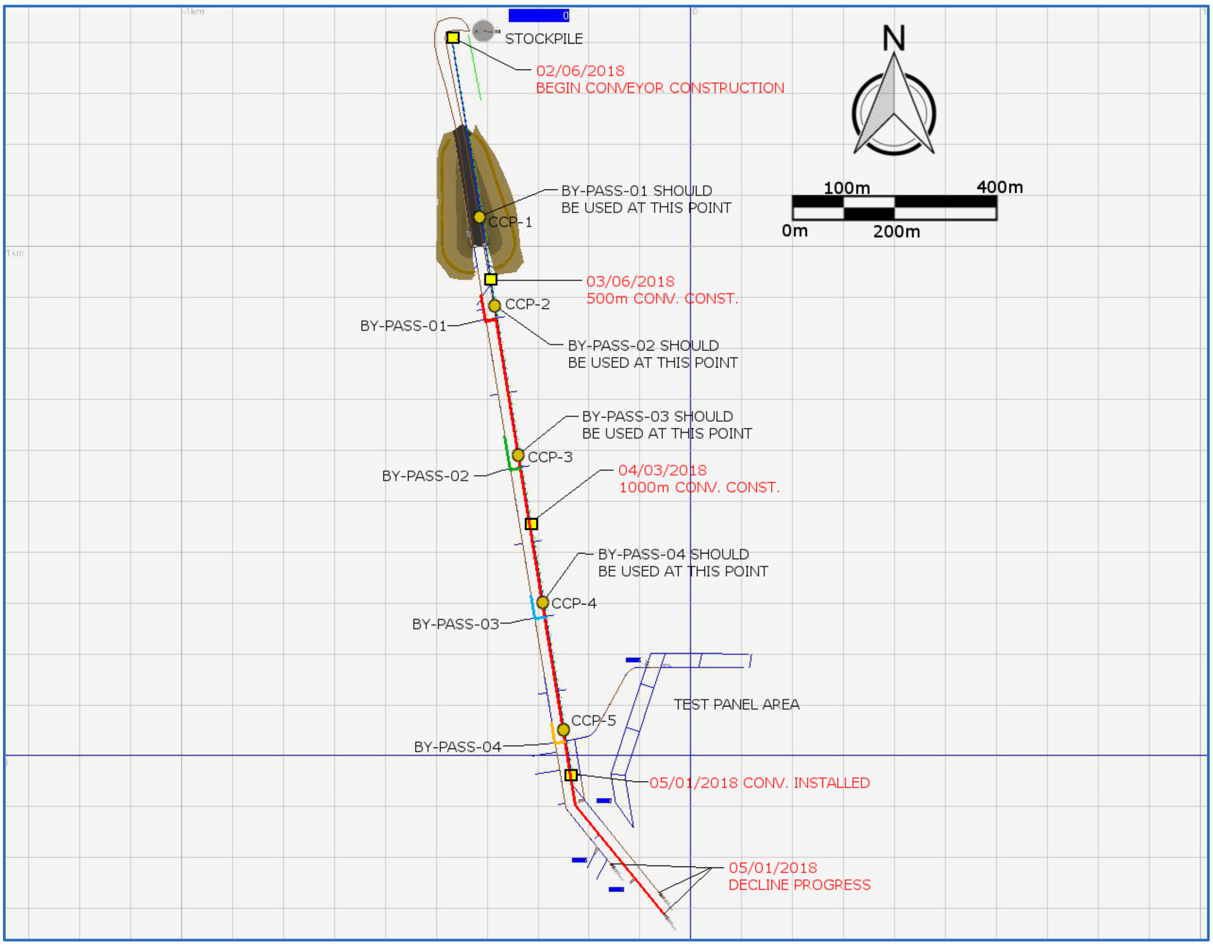


Figure by Kamoa Copper SA, 2015.

16.4.4 Mining Schedule

16.4.4.1 Development and Construction Schedule

The development schedule focuses on establishing necessary mine services and support infrastructure to set up the initial production mining areas, to add an additional 1.6 Mtpa to the Kakula-Kansoko processing plant, and then to ramp up to 6.0 Mtpa ore production once Kakula production decreases. The full production schedule will be based on a 360-day calendar that will be sustained for 14-years with a 37-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 16.38 summarises the LOM development and production quantities.

Table 16.38 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 panel 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 in the initial years of production.

16.4.4.2 Mine Development Plan and Scheduling

For primary development, the rates in Table 16.39 were calculated using first principles. Cycle inputs were obtained from various sources such as original equipment manufacturer [OEM], external consultants, specialists, and compared with Stantec inputs. The cycles were updated accordingly following team discussions.

Table 16.39 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 variances, which is within the required accuracy of the study. Only 4.0 m and 6.0 m high drift sizes in the ≥30° to 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.

16.4.4.3 Preproduction Development Schedule

The initial development in the ramp-up period ramping up to 1.6 Mtpa requires some waste development mined is the Kansoko Centrale. 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.

16.4.4.4 Life-of-Mine Development Schedule

The development schedule beyond the initial ramp up to 1.6 Mtpa 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 16.42 illustrates the LOM development schedule.

Figure 16.42 Life-of-Mine Development Schedule

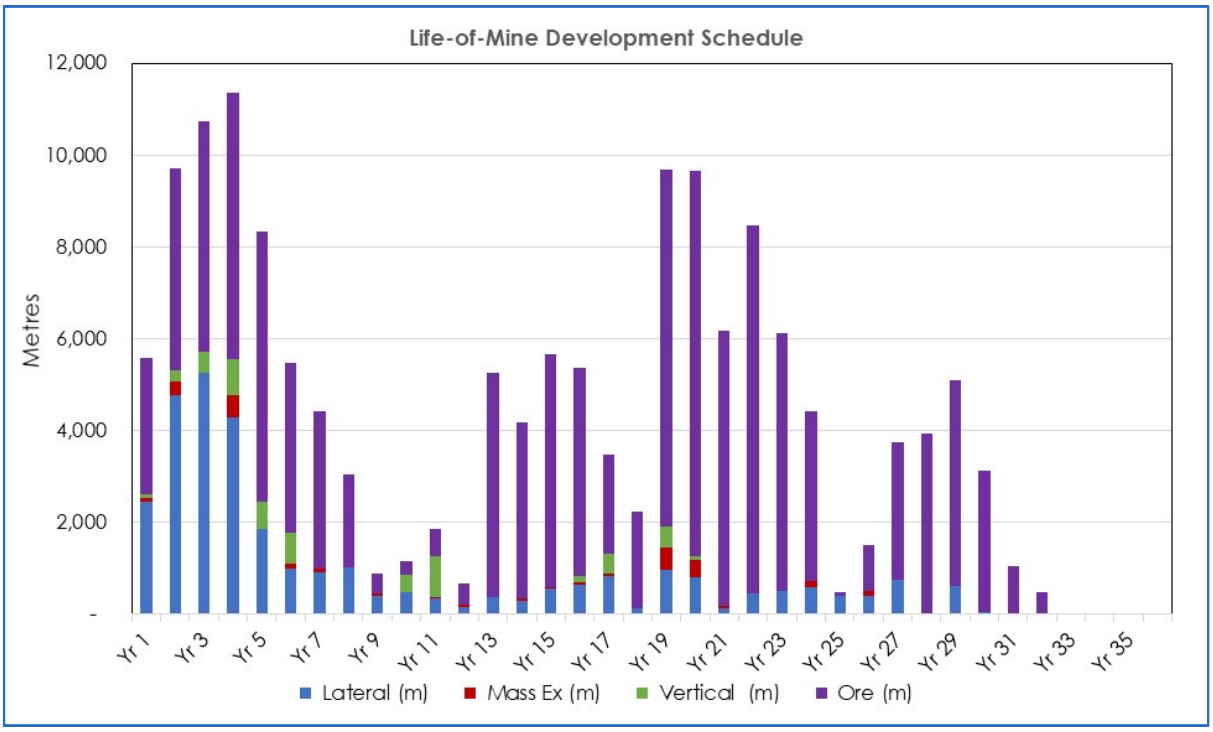


Figure by OreWin, 2020.

16.4.4.5 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 for a production rate of 1.6 Mtpa. Then after the Kakula production declines Kansoko will ramp-up to 6.0 Mtpa ore production and associated development waste. The full production schedule will be based on a 360-day calendar that will be sustained for 14-years with a 37-year LOM.

The following criteria were established for the targeted resource, to support the overall tonnage requirements from the Kamoa deposit. Table 16.40 details the targeted annual tonnages for the overall production requirements to meet 1.6 Mtpa initial and the 6.0 Mtpa final production rate.

Table 16.40 Production Schedule Criteria

Criteria	Details	
Initial and Ramp-Up	2.3 Mt	
Full Production	6.0 Mtpa	
Extraction / Recovery	75%	
Production Schedule	**Years**	**Tonnes**
Initial Production Mining (Year-1)	1	290,000
Ramp-Up (Year-2)	1	760,000
Ramp-Up (Year-3)	1	1,200,000
Production (1.6 Mtpa)	14	25,000,000
Full Production	14	82,000,000
Production (prior to ramp down)	31	109,250,000

The Kansoko development schedule initially is 1.6 Mtpa creating a combined 7.6 Mtpa for the Kakula-Kansoko 2020 PFS. The Kansoko schedule then focuses on establishment of necessary mine services and support infrastructure to set up the initial production mining areas and ramp-up to 6.0 Mtpa ore production and associated development waste. 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. Figure 16.43 illustrates LOM schedule and grades. The mine production schedule is detailed in Table 16.41.

Figure 16.43 Life-of-Mine Schedule and NSR

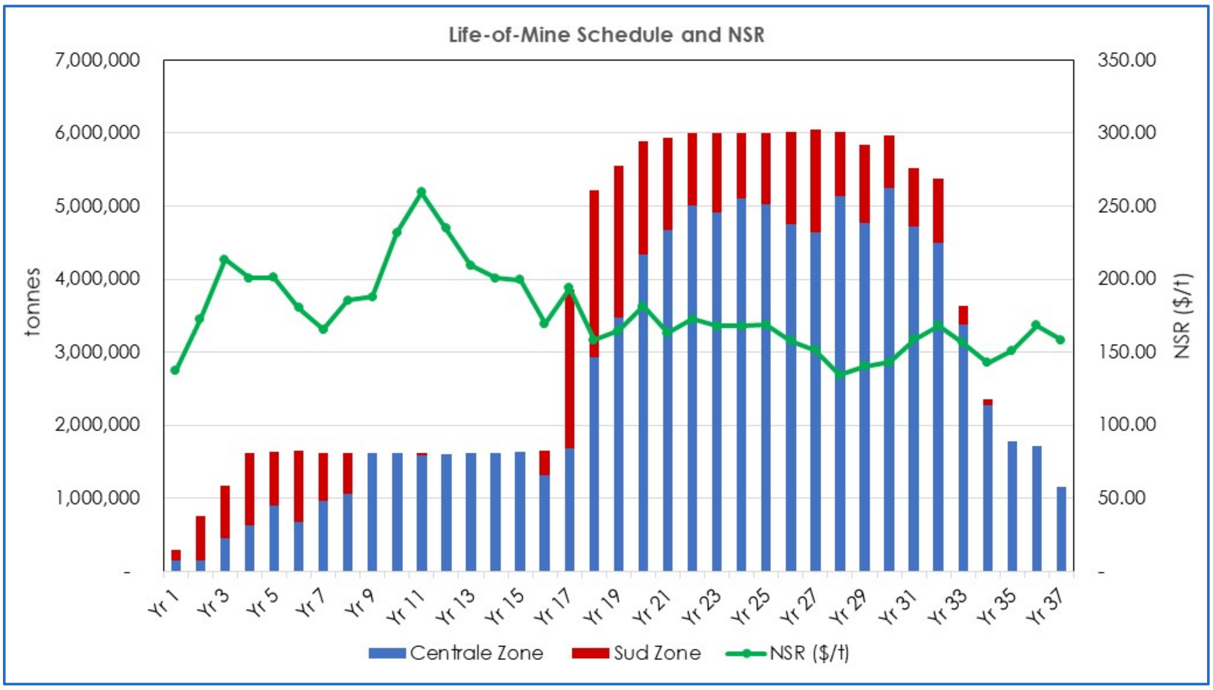


Figure by OreWin, 2020.

Table 16.41 Kansoko Mine Production Schedule

Description	Unit	Total	Project Time (Years)																		
			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
Room-and-Pillar Ore Mined	(kt)	3,397	–	347	717	941	732	660	–	–	–	–	–	–	–	–	–	–	–	–	–
	(% Cu)	5.29	–	5.49	5.97	5.17	5.60	4.28	–	–	–	–	–	–	–	–	–	–	–	–	–
Controlled Convergence Room-and-Pillar Ore Mined	(kt)	111,120	–	–	–	49	–	325	1,052	860	1,521	1,615	1,564	1,598	1,612	1,612	1,627	1,308	3,622	5,013	4,833
	(% Cu)	3.81	–	–	–	2.78	–	2.88	3.96	4.25	4.25	5.20	5.90	5.28	4.73	4.53	4.49	3.93	4.46	3.65	3.85
Ore Development Ore Mined	(kt)	10,665	289	414	450	635	898	673	562	762	95	–	56	12	–	5	2	336	231	205	719
	(% Cu)	3.34	3.15	2.56	2.64	3.56	3.45	4.30	3.44	4.15	4.26	–	2.96	3.07	–	2.33	3.13	3.58	3.08	2.96	3.11
Total Ore Mined	(kt)	125,182	289	761	1,168	1,625	1,630	1,658	1,614	1,622	1,616	1,615	1,620	1,610	1,612	1,617	1,629	1,645	3,853	5,218	5,552
	(% Cu)	3.81	3.15	3.89	4.69	4.47	4.42	4.01	3.78	4.21	4.26	5.20	5.80	5.26	4.73	4.53	4.49	3.86	4.38	3.62	3.75

Description	Unit	Total	Project Time (Years)																		
			20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	36	37	38
Room-and-Pillar Ore Mined	(kt)	3,397	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
	(% Cu)	5.29	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Controlled Convergence Room-and-Pillar Ore Mined	(kt)	111,120	5,105	5,402	5,300	5,493	5,680	5,998	5,927	5,777	5,712	5,439	5,699	5,423	5,327	3,628	2,346	1,772	1,719	1,159	–
	(% Cu)	3.81	4.20	3.76	4.05	3.87	3.86	3.84	3.62	3.46	3.11	3.20	3.28	3.63	3.82	3.54	3.26	3.43	3.80	3.58	–
Ore Development Ore Mined	(kt)	10,665	784	534	700	514	321	8	89	270	304	394	262	92	48	–	–	–	–	–	–
	(% Cu)	3.34	3.61	3.25	2.91	3.26	3.14	2.78	2.92	3.30	2.61	3.17	3.00	2.88	3.54	–	–	–	–	–	–
Total Ore Mined	(kt)	125,182	5,889	5,936	6,001	6,007	6,001	6,007	6,016	6,048	6,016	5,833	5,962	5,516	5,375	3,628	2,346	1,772	1,719	1,159	–
	(% Cu)	3.81	4.12	3.72	3.92	3.82	3.82	3.84	3.61	3.45	3.09	3.20	3.26	3.61	3.82	3.54	3.26	3.43	3.80	3.58	–

Totals may not add up due to rounding.

16.4.5 Underground Infrastructure

16.4.5.1 Ventilation

Diesel particle matter is the main driver for establishing airflow requirements for the underground openings. These requirements are 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, is used 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 determine mine air cooling and refrigeration requirements. The strategy is to ventilate the mining sections with flow-through ventilation and avoid of recirculation/reuse of air. Main service and conveyor declines provide fresh air, while a ventilation raise near the bottom of the main declines (Vent Raise No. 1) is used as an exhaust shaft. Fresh air from the declines splits into two major development fronts — one supporting Centrale and the other Sud. Bulkheads, ventilation curtains, seals, pillar sections, and booster fans 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 developed blind to every other cross-cut, where the cross-cut used to establish a loop for the next blind segment. When the loop is created, the service decline is used as the return airways and the conveyor decline provides the intake air. During the development of the blind headings, an exhaust overlap system is used for maximum performance and safety.

The initial exhaust ventilation raise (Vent Raise No. 1) located at the bottom of the main declines provides ventilation to Centrale and Sud primary development headings. The main twin declines provide fresh air, and the return flow is exhausted through the ventilation raise.

As the mine life progresses, additional intake and exhaust ventilation raises are 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 heading" to the mining areas will deliver fresh air as a flow-through system. Two primary service drifts and one primary conveyor drift are driven blind to every other cross-cut, where the cross-cut is 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 are the primary ventilation route for the panel. These are developed following the contour elevation of the ore, orientated close to the strike. These are twin headings with cross-cuts between the drifts. During the development of the secondaries, one heading is the intake and the other functions as an exhaust.

Typical mining direction within the controlled convergence room-and-pillar panels begin at the extremities. Rooms and belts are mined adjacent to the secondary at the extremities of the panel to establish flow-through ventilation. The typical ventilation circuit flows through the secondary drifts into the active mining area once the connection is established. Fresh air flushes over the mining face due to the negative pressure from the exhaust side which is connected to an exhaust raise. Air distribution within the rooms is 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, 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 kW, 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.

16.4.5.2 Mine Air Cooling Facilities

Refrigeration will be required to provide sufficient cold air and to ensure that the development and panel exhaust temperatures remain within design parameters (i.e. average development and panel exhaust wet bulb temperatures of 28.5°C). A first-order comparison of alternative refrigeration systems, notably underground refrigeration installation and surface ice makers, showed 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 bulk air cooling (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.

16.4.5.3 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 days/yr. LHDs and haul trucks will transport the rock from the working headings. While production is at the 1.6 Mtpa rate ore will be moved to surface with haulage trucks. Once the production ramps up to 6.0 Mtpa 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 trackless operation designed with a 6.0 Mtpa capacity for ore and waste handling. For the 6.0 Mtpa production rate 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 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.

16.4.5.4 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 6.0 Mtpa Ramp up development phase of work, 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 H 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 16.44 indicates an overall layout of the conveyor belt network.

Figure 16.44 Conveyor Network System

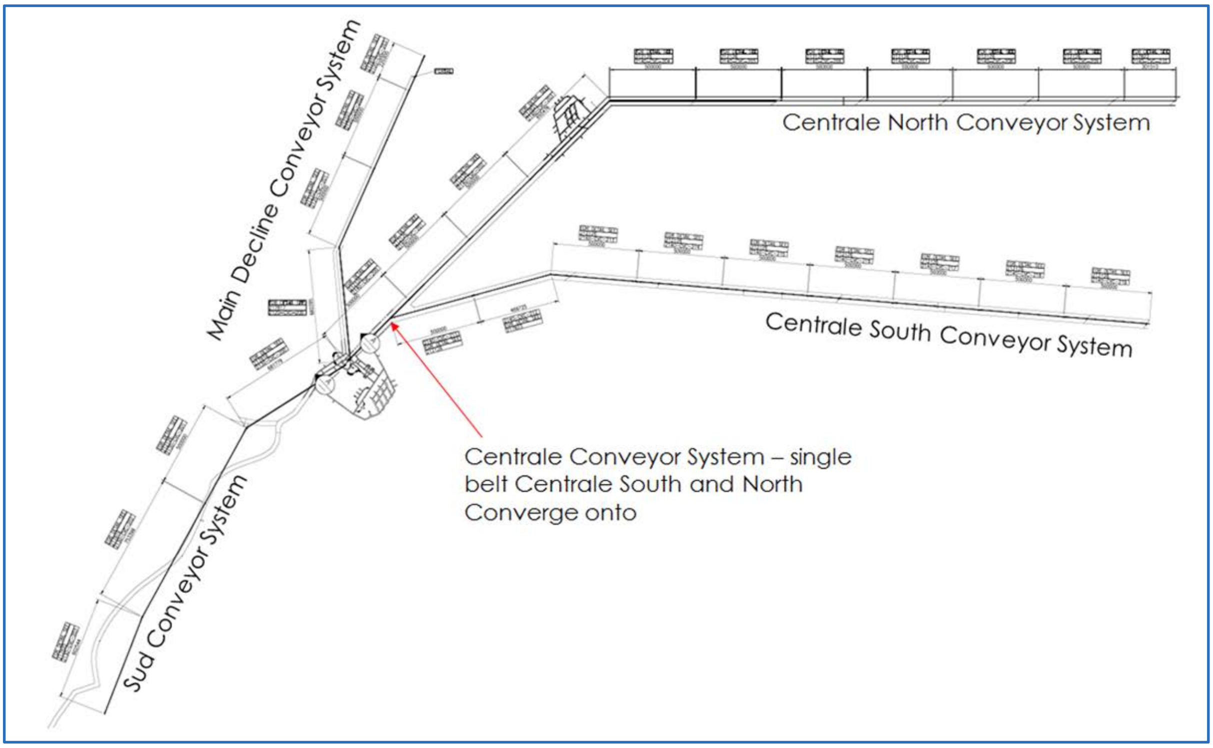


Figure by Stantec, 2017.

All the decline conveyors are designed to convey the total mine ore production of 6.0 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 L x 1,500 mm W 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/day (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/day (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.

16.4.5.5 Workshops

Major mobile equipment will remain underground for the duration of the machine’s life cycle and will be serviced and 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 is 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 is 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 such as drill rigs, bolters, and LHDs are serviced and maintained (minor repairs) at satellite workshops. All vehicles revert to the main workshop for major services and repairs. Trucks hauling waste to surface as well as UVs are maintained in the surface workshop. The surface workshop is also equipped for rebuilds.

The bulk of tyres are stored on surface with a minimum quantity of tyres stored underground in the allocated bay. Tyre repairs and fitting to the required rims is undertaken on surface and transported to and from the workshops daily as required. The tyre bay is equipped with a 5-tonne overhead crane.

A multi-purpose vehicle (MPV) equipped with a tyre handler will 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.

16.4.5.6 Fuel and Lubricant Distribution

The mine is 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 refuelling pumps will be constructed on surface, near the portal. These are used initially until the facilities at the main workshop are completed. The initial surface installation is used for LOM for the refuelling 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 is piped down, through a dedicated borehole, from surface. The pipe column to the main workshop fuel storage area is an “energy dissipation” pipe to prevent high-flow velocities and pressure build-up. The diesel is batch fed to the underground storage tanks at the main workshop. Fuel is batch pumped from the tanks at the main workshop to the tanks at the Centrale workshop via a 50 mm diesel pipe. Refuelling stations are available at both the main and Centrale workshops. Fuel is distributed to working sections from the Centrale and main refuelling stations via diesel bowser cassettes. These cassettes refuel slow-moving or captive equipment such as drill jumbos, bolters, pillar scrapers, and shotcrete spray units. Tier 2 diesel engines are 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 lubricants will be transferred to surface workshops via dedicated pipelines and transported underground in lubrication cassettes. The underground lubrication cassettes have dedicated storage areas. Surface storage tanks are designed with sufficient lube storage capacity to operate the mine for approximately 30-days (per grade of oil).

Waste oil and other fluids are collected in designated cassettes. The waste oil is discharged from the collection container into a bulk tank, which, when full, is transported out of the mine.

16.4.5.7 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 uses the two-component emulsion system, consisting of a base product and a sensitiser combined at the face.

Each type of explosive is transported underground separately and via different methods. Class 1 explosives, which include Explosive 1.1B and Explosive 1.1D, is transported underground using purpose-built explosives cassettes. Oxidiser and base emulsion are piped down from surface into storage tanks underground; distribution into the mining areas is via emulsion cassettes.

AEL Mining Services 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 store explosives near working areas or in the Centrale emulsion storage magazine.

16.4.5.8 Concrete and Shotcrete Facility and Distribution

A LOM concrete and shotcrete batch plant built on the surface delivers wet cementitious product. A positive displacement pump connected to a pipeline will be used to transfer the cementitious mix from the batch plant to a borehole to underground mine. In the underground mine, a transmixer and agicar truck transports the shotcrete from the borehole to the shotcrete placer pump truck and discharges the load into the hopper of the shotcrete placement pump.

16.4.5.9 Compressed Air System

The compressed air system does not include a mine-wide reticulation system from the main surface facility. Piping is provided from the compressor station on surface to the surface facilities, main underground workshop, and main refuge station. Permanent compressed air piping is routed from surface through the decline to the underground workings.

Three 1,700 cfm compressors are located on surface to supply air to the surface infrastructure and underground areas. The compressor is connected to the mine’s emergency power supply, ensuring that compressed air is supplied to the main refuge chamber during power outages.

Compressed air piping installed during the development in the primary declines and extends into the underground maintenance workshop and into the permanent underground refuge chamber. General use compressed air does not extend to other sections of the mine; however, consideration is given to extending piping 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 are sized for the equipment they 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.

16.4.5.10 Water Management

Water management from surface to underground consists of managing service, potable, and fire water systems. The water management from underground to surface consists of the mine dewatering system, which includes the production return water system and main pump stations.

Potable water required for use by underground personnel is provided from the potable water surface supply. The potable water storage supplies the surface facilities as well as the underground mine. A potable water pipeline is routed from surface through the decline to the underground infrastructure. Pressure reducing stations are installed as required. Potable water is provided 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 is routed from the surface through the decline to the underground, and to future development, and remains in development for later use. Similarly, MSW pipelines are installed in mine openings driven in ore for panel development and are left for ore production needs. These pipelines are removed from each panel for re-use after production is completed. Pressure regulators manage the increase in static water pressure created as the declines progress. Pressure-reducing stations have redundant regulators for service and maintenance should issues occur.

16.4.5.11 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 is 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 is 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 are fitted with dual suction and designed to supply dedicated firewater for a duration of 90 minutes at maximum flow.

Fire water reticulation pipe work is SANS 62 MED WT galvanised and banded pipe. All fittings and flanges are Class 16. All isolation/section valves are Underwriters Laboratories (UL) listed and Factory Mutual (FM) approved. The pipe is installed in the conveyor declines and included in the conveyor designs and costs. The conveyors located in the main decline are installed in the intake and equipped with a fire line across the entire length of the belts.

Fire hydrants are fed by the fire water column and placed no further than 60 m apart in the required areas.

All substations and MCCs have both a smoke detection system in the room as well as a VESDA (Very Early Smoke Detection Apparatus) in the cabinets. Each installation has its own panel for remote status monitoring via potential free contacts. Each installation is 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 also contain potential free contacts used for the shutting down of associated equipment (e.g. main incomer, air conditioning system).

Flame detectors are placed at strategic locations to detect a moving fire in its incipient stage. The detection system will initiate belt shutdown and activate the solenoid on the associated deluge valve. Each detection system has its own control panel with potential free contacts for belt shutdown as well as remote monitoring of fire and fault signals.

16.4.5.12 Materials Handling Logistics and Storage

Materials, equipment, and mining supply items are delivered by road to the mine site warehouse located at the surface. The mine site warehouse manages and sources services for both the process plant and Kakula and Kansoko mining operations.

Designated underground storage areas are located throughout the mine and typically in proximity to the point of use. Storage areas designated for infrastructure support such as explosive magazines, fuel and lubricates, and warehouse items in transit, have permanent ground support including shotcrete. These areas have concrete floors and lighting.

Mining supplies managed and sourced from the surface mine site warehouse will be kept in laydown areas close to the mining operations. The main laydown area is designed as a drive-through.

16.4.5.13 Refuge Stations

Refuge stations are required to house underground mining personnel in a secure, hazard-free location during emergency conditions. A constructed or modular-style refuge station is located near the underground maintenance workshop area. The workshop refuge station is serviced with either compressed air from surface or equipped with a self-contained breathing systems. Portable refuge stations are used 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 are sized to meet the capacity requirements for the area.

16.4.5.14 Toilet System

Underground sewerage comprises of 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 manoeuvrable. Each unit is fitted with a sump and pumped empty into a sewage tank mounted on a cassette carrier. This tank is transported to surface and emptied into a sewerage disposal system on surface.

16.4.5.15 Electrical Substations and Power Distribution

Power is distributed at 11 kV to the underground mine switchgear from two surface feeder breakers, for redundancy. The underground switchgear is contained in an E-room and has separate feeder breakers feeding major mine areas for isolation purposes and to minimise large connected loads to each feeder. Each feeder supplies multiple mine power centres, which 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 is distributed to Centrale North, Centrale South, and Sud through the primary development headings. The power also supplies 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.

16.4.5.16 Communication, Controls, and Automation Systems

The backbone for the communications system is based on a redundant fibre network. This system supports all voice and data communication requirements for the Project. Radio communications for the mine is provided over a leaky feeder system, which is distributed throughout the entire mine for communication purposes, incorporating hand-held and fixed radios. This is 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 is 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 are installed at each rock breaker, conveyor transfer point, and pump station. Fibre is installed for monitoring of the power system and control for conveyors, pumps, and rock breakers. A fibre allowance includes capacity for ventilation-on-demand, if required.

Upon entering the mine site through the surface access-controlled complex, mining personnel proceed to the Change House and Lamp Room. Access into and out of the mine is controlled by means of an electronic tag-in/tag-out system integrated into the cap lamps, which is monitored in the Control Room.

16.4.6 Mining Equipment

The criteria considered in equipment selection includes 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 are based on the LOM production schedule associated development. The schedule for equipment purchases and replacement are based on a rebuild and replace cycle. No equipment is replaced within 2-years of the end of the LOM.

16.4.6.1 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 clean-up 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 16.42.

Table 16.42 Mobile Equipment List

Description	Yearly Maximum Required	Number of Units to Purchase or Replace	Number of Rebuilds
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

16.4.6.2 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 16.43 summarises the main fixed equipment for the Kansoko mine design.

Table 16.43 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		

16.4.7 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 such as 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.

Competent mining crews, in particular mobile production equipment operators, are essential in safely achieving production targets. A training department for both mining and engineering is allowed for in the labour complement and a training facility available on surface for technical training. Practical training is carried out underground, on the job, where final assessment for certification is done. Recruitment of local labour will require training to be conducted in French and Swahili.

Direct and indirect labour 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 work 360 days/year. Owner capital work and production are accomplished in 360 days/year.

17 RECOVERY METHODS

17.1 Kakula Concentrator Plant

17.1.1 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 process plant design was undertaken by DRA Projects (PTY) Ltd (DRA Projects).

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.

17.1.2 Kakula Concentrator Basis of Design

The Kakula concentrator is designed to process a maximum throughput of 7.6 Mtpa and includes all ore processing requirements from the bottom of the run of mine stockpile through to final concentrate dispatch and final tailings disposal. The Kakula concentrator design criteria are shown in Table 17.1.

Table 17.1 Kakula Concentrator Plant Design Criteria

Design Parameter	Unit	Nominal Value	Design Value
Annual Surface Crushing Circuit Feed	Mtpa	7.6	8.0
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	Mtpa	6.0	7.6
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

17.1.3 Plant Design and Process Description

The concentrator consists of the following:

- A shared crushing and screening module.
- Two identical high-pressure grinding rolls (HPGR) milling circuit.
- Two identical flotation circuits, complete with concentrate regrind circuits.
- Shared concentrate thickening, filtration, bagging and loading systems.
- Two identical tailings thickening circuits.
- Shared tailings disposal and backfill plant feed systems.
- Shared utility systems for air, water and reagents.

A high-level block flow diagram of the Kakula concentrator plant is Figure 17.1.

Figure 17.1 Kakula Block Flow Diagram

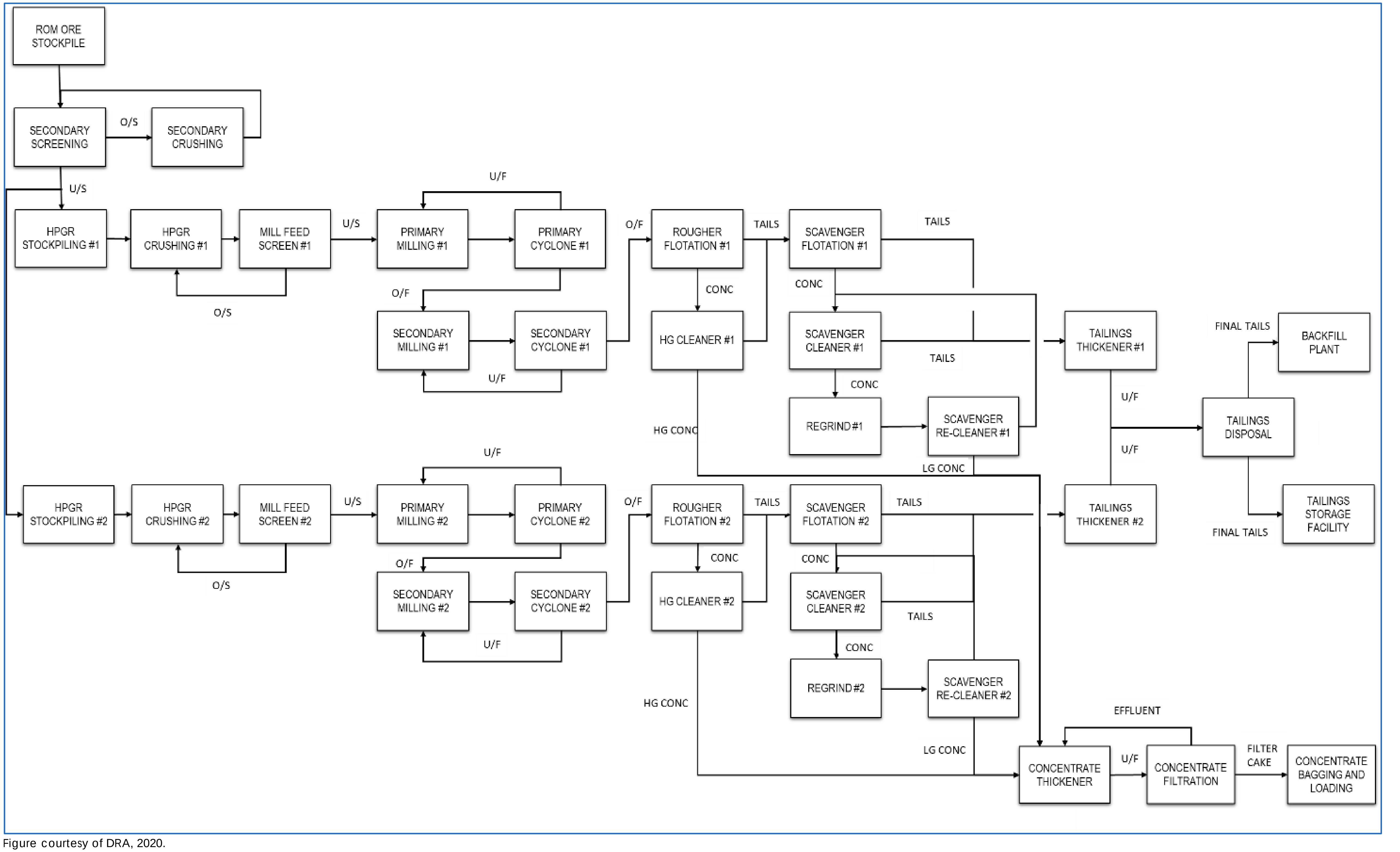


Figure courtesy of DRA, 2020.

A description of the main components of the process follows.

17.1.3.1 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.

17.1.3.2 Crushing and Screening

The secondary screening feed conveyor transfers material from the ROM stockpile, together with secondary crusher product, to the 285 t secondary screening feed bin. The material is screened at 50 mm using two 3.6 x 7.0 m, dual deck, vibrating screens. The secondary screen oversize material, roughly 60% of the screen feed, is conveyed to the secondary crushing circuit for size reduction, while the secondary screen undersize material reports to either one of the two HPGR feed stockpiles via the secondary screen undersize transfer conveyor.

The secondary screen oversize material reports to the 225 t secondary crushing feed bin, via the secondary crushing feed conveyor. The material is extracted at a controlled rate using dedicated feeding systems to feed three 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} 195 mm to P_{80} 55 mm.

The secondary cone crusher product is conveyed to the secondary screening feed bin via the secondary screening feed conveyor.

Tramp iron removal systems are included on the secondary crushing feed conveyor. Provision is made for dust suppression at the screening and crushing buildings. Process cameras are provided at the secondary screening building for monitoring.

17.1.3.3 HPGR Stockpiling

The secondary screening undersize product is conveyed to the actuated, HPGR storage feed splitter chute arrangement where the material can be split between the two HPGR feed stockpiles or directed to either one of the two stockpiles depending on stockpile levels via the HPGR feed stockpile conveyors.

Each HPGR feed stockpile is designed to store a live capacity of 5,000 t. The material is extracted from each of these stockpiles, at a controlled rate via two variable speed belt feeders, which discharge the material onto dedicated HPGR feed bin transfer conveyors.

Provision is made for a shared dust control system in the area, as well as various process cameras for monitoring purposes.

17.1.3.4 HPGR Crushing

The HPGR crushing circuits will consist of two identical modules.

The secondary screening undersize product is extracted from the HPGR feed stockpile at a controlled rate using two variable speed belt feeders, which discharge the material onto the HPGR feed bin transfer conveyor. The HPGR feed bin transfer conveyor transfers the material onto the HPGR feed bin conveyor, where the secondary screen undersize product is combined with the primary mill feed screen oversize recycle stream.

The combined HPGR feed material reports to the 225 t HPGR feed bin, via the HPGR feed bin conveyor. The material from the HPGR feed bin is extracted at a controlled rate using variable speed vibrating feeder which discharge onto the HPGR feed conveyor. Tramp iron removal systems are included on the HPGR feed conveyor in the form of a tramp iron removal magnet followed by a metal detection unit.

The HPGR unit, a Polycom HPGR 17/12-5, is fitted with 2 x 1,200 kW drives to achieve a size reduction from F_{100} 50 mm. Provision is made for dust suppression at the HPGR building.

HPGR crushed ore is conveyed to the primary mill feed screen for closed circuit classification at 8 mm. The primary mill feed screen - a dual deck 3.6 m x 6.1 m vibrating unit, is utilised for primary mill feed classification. The primary mill feed screen oversize product (plus 8 mm) is collected on the HPGR feed bin conveyor and recycled to the HPGR circuit for size reduction. The screen undersize material (minus 8 mm) gravitates to the primary mill feed hopper where it combines with the primary mill classification cyclone underflow.

17.1.3.5 Primary Milling

The primary milling circuit will consist of two identical modules. Each module comprises of a 20'Ø x 32' EGL, overflow discharge ball mill (installed with a 7000 kW VSD) operating in closed circuit with a cyclone cluster consisting of 12 x 500 mm diameter units.

The primary mill feed screen undersize material (–8 mm) gravitates to the primary mill feed hopper where it combines with the primary mill classification cyclone underflow.

The primary mill slurry gravitates to the 150m^3 primary mill discharge sump, via a trommel 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} 140 µm, reports to the secondary mill discharge sump as feed.

Addition of 70 mm high chrome steel balls is affected using a magnet and sputnik arrangement, to load the media to the primary mill feed hopper via the primary mill feed screen underpan.

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 trommel screen (scats) reports to dedicated scats conveying systems.

The design allows for dedicated mill relining machines to each primary mill to facilitate with mill relining. Provision is made for process cameras at the primary mill feed screen areas, for monitoring purposes.

17.1.3.6 Secondary Milling

As per the primary milling circuit design, the secondary milling circuit consist of two identical modules. Each module comprises of 20'Ø x 32' EGL overflow discharge ball mills (installed with a 7,000 kW VSD), operating in closed circuit with a cluster of 16 x 350 mm diameter classification cyclones.

The primary milling classification cyclone overflow products report to the 150 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, 400 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 is affected using a magnet and sputnik arrangement, to load the media to the secondary mill feed hopper. Dosing of reagents are via dedicated dosing funnels.

Provision is made in the design for spillage collection and pumping systems, as well as a process camera at the rougher flotation feed tank area.

17.1.3.7 Rougher/Scavenger Flotation

The rougher flotation circuit consist of two identical modules, each comprising of a single bank of eight 300 m^3 mechanically agitated, forced air flotation tank cells in series, to produce two concentrate products. 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 m^3 high grade rougher concentrate sump, via the high-grade cleaner feed trash removal screen. The high-grade rougher concentrate 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 six cells and gravitates to the 22 m^3 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 and third cells' concentrate product to either the high or the low-grade product, as required. Provision is made for dosing of collector, promoter, and frother to the second cell's feed box.

The scavenger tailings product gravitates to the 45 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.

Provision is made for process cameras at the high-grade feed linear screens for monitoring purposes.

17.1.3.8 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 using a fixed speed, duty / standby pumping system. The design includes an online grade analyser for monitoring of the high-grade concentrate grade. Provision is made for froth washing water to the Jameson cell in the form of filtered water.

The tailings from the high-grade cleaner cell gravitates to the 12 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 fixed 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 tailing's sump via a vertical spindle pump.

17.1.3.9 Scavenger Cleaner Flotation

The scavenger cleaner flotation circuit consist of two identical modules, each comprising of a single bank of 6 x 160 m³ 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 12 m^3 scavenger cleaner concentrate sump, from where it is pumped to the concentrate regrind classification cyclone via a variable speed, duty/standby pump system.

The scavenger cleaner tailings gravitate 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.

17.1.3.10 Concentrate Regrind Milling

The concentrate regrind milling circuit consist of two identical modules, each comprising two high intensity 355 kW SMD regrind mills, operating in open circuit with a cluster of 14 x 100 mm diameter cyclones.

The scavenger cleaner concentrate product (P_{80} 30 µm) from each flotation modules is 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 22 m^3 regrind mill product sump, via a trash removal screen.

The cyclone underflow product (P_{80} 80 µm) gravitates to an equal flow splitter box where it is split as feed to each of the two regrind mills for regrinding to produce a product at 80% passing 10 µm. After trash removal screening, 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 regrind mills. Process cameras will be installed at each scavenger recleaner feed trash screen for monitoring purposes.

17.1.3.11 Scavenger Recleaner Flotation

The scavenger recleaner flotation circuit consist of two identical modules, each consisting of a single low entrainment Jameson flotation cell.

The concentrate regrind circuit product is pumped to the scavenger recleaner cell pump sump (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 5 m^3 scavenger recleaner concentrate sump from where it is pumped to the concentrate thickening circuit via an online grade analyser.

The tailings from the scavenger recleaner cell gravitates to the 22 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 fixed speed duty/standby pumping systems.

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.

17.1.3.12 Flotation Tailings Thickening

The flotation tailings from each module is pumped to dedicated, 38 m 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 are thickened to an underflow product containing 55% solids (w/w), before being pumped to the backfill feed surge tank, via a two-stage sampling system, using variable speed, duty/standby pumps.

Tailings thickener overflow products gravitate to dedicated 3 km^3 process water tanks 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.

17.1.3.13 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 mechanically agitated, 200 m^3 backfill feed surge tank. The thickened flotation tailings are pumped to the backfill circuit using dedicated variable speed pumping trains per backfill module.

Excess tailings from the backfill feed surge tank overflows to the 100 m^3 final tailings sump where dilution water is added to obtain densities suited for long distance pumping. The diluted tailings are then transferred to the TSF. The final tailings disposal system consists of three pump trains each comprising four high pressure centrifugal pumps in series, delivering slurry to the TSF via dedicated pipelines.

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 tailing's disposal area gravitate to the spillage collection sump from where it is pumped to the final tailing's sump using a submersible pump.

17.1.3.14 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 two-in-one sampling system. The sampled stream gravitates to the concentrate thickener feed box 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 55% 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.

17.1.3.15 Concentrate Filtration Feed

The thickened concentrate is pumped to either one of two 500 m^3 mechanically agitated filtration feed tanks. 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 submersible pump.

17.1.3.16 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 to dedicated, reversible, 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.

Provision is made for a reloading point at the tail end of the concentrate loadout conveyor, to introduce emergency stockpile material to the bagging system feed.

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.

17.1.3.17 Concentrate Bagging and Loading

The design of the concentrate bagging and loading facility is based on a 24 hr operation per day. operation. Provision is made in the design to load 50% of the 7.6 Mtpa produced concentrate into bulk trucks, and the remainder into bags. During phase 1, 100% of the concentrate can either be bagged or loaded into bulk trucks.

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 two modules, each fed from a dedicated 17 m^3 storage bin and associated belt feeders. The first module is equipped with a bagging carousel, while the second unit is equipped to load into bulk trucks on a weight basis.

The bags will be sampled manually after which composite samples of each lot will be formed and assayed prior to dispatch.

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.

17.1.3.18 Air Services

Low pressure blower air to each of the forced air tank flotation cells are supplied by five variable speed multistage centrifugal air blowers operating as four 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 air required on the concentrator plant is produced by three low pressure instrument air compressors (two running, one standby) at 1,300 kPa. Air produced by the instrument air compressors passes through duty/standby air filtering and drying systems before being stored in two 10 m^3 instrument air receivers (one per concentrator module). Due to the high instrument air requirement from the milling, the design further includes for two additional 10 m^3 instrument air receivers – one located in each milling circuit. The instrument air is stored at 1,300 kPa and distributed at 750 kPa.

Drying air to the concentrate filter units is supplied by two 1,300 kPa compressors (drying pressure at 1,100 kPag) in a duty/standby configuration and stored in a single 20 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 2,000 kPa compressors (pressing cycle pressure at 1,600 kPag) 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.

17.1.3.19 Water Services

The water circuit design for the concentrator circuit consist of three separate systems, i.e. process water, filtered water and potable water.

The two 3 km^3 process water tanks are fed by any excess concentrate thickener effluent, TSF return water, and both tailings thickener overflow products. 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 supply to the backfill plant.

The sand filter plant is fed by a mixture of TSF return, backfill effluent, underground water, and raw water. Filtered water product from the sand filter plant is pumped to the 1 km^3 filtered water tank, from where it is distributed across the concentrator plant for use as gland seal water, dust suppression, mill cooling water and Jameson cell spray water.

The gland seal water pump circuit is designed as a separate system due to higher pressure requirements and includes dedicated pressure-controlled gland seal water ringmains to each of the milling-flotation modules, fed from duty/standby gland seal water pumps. Other raw water is distributed to the required points at a controlled pressure using the duty/standby concentrator raw water pumps.

Potable water from the Kakula site potable water tank is distributed to the required points and used for human consumption, reagent mixing and safety showers. Fire water to the concentrator is supplied from the common fire water system, located at the mining area.

17.1.3.20 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 25 m^3 mechanically agitated collector mixing tank, where it is diluted with potable 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 is pumped 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. Based on nominal consumption rates, two batches are required per day.

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, as well as flash back arrestors on each storage tank.

17.1.3.21 Promoter Make-up and Dosing

AERO 3477 is used as promoter in the flotation circuit and is delivered as a 50% (w/v) liquid in 1 t flobins.

The design includes for duty/standby promoter transfer pumps to transfer the liquid from the 1 t flobins to the mechanically agitated, 10 m³ mixing tank where it is diluted with potable 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 is pumped to the 20 m³ promoter dosing tank using the same transfer pumps. The reagent is distributed to the designated dosing points using a duty/standby peristaltic pump system to feed a pressure controlled ringmain.

Based on nominal consumption rates, one batch is required every day. 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.

17.1.3.22 Frother Dosing

SF522 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.

17.1.3.23 Flocculant Make-up and Dosing

The design allows for use of SNF 910SH as flocculant at the concentrate and both tailings thickeners. The flocculant is delivered as a powder in 750 kg bags.

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 either one of the two 100 m³ mixing/dosing tanks, via vendor supplied wetting systems.

Provision is made for potable water addition to 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 and filtered water.

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.

17.1.3.24 Coagulant Make-Up and Dosing

The design allows for use of SNF 45 VHM as flocculant at the two tailings thickeners. The coagulant is delivered as a powder in 750 kg bags.

Bags are manually hoisted and discharged into the vendor supplied coagulant bulk bag bin receiver. A screw feeder is used to transport the dry reagent into either one of the two 100 m³ mixing/dosing tanks, via vendor supplied wetting systems.

Provision is made for potable water addition to both the mixing/dosing tanks for dilution to a dosing 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.

Spillage produced in the coagulant area is collected in the flocculant spillage sump and pumped to the final tailing's disposal sump via a vertical spindle pump.

17.1.4 Concentrator Services Requirements

Table 17.2 lists the estimated projected water, consumables and power requirements for the concentrator.

Table 17.2 Projected Concentrator Water, Power, and Consumables

Item	Description	Consumption per tonne of Plant Feed	Annual Requirement
Power	Electricity	41.8 kWh/t	251 GWh
Water	Raw make-up	0.20 m³/t	2,400 ML
Reagents	Frother	203 g/t	1,542 t
	Collector	310 g/t	2,355 t
	Promotor	56 g/t	425 t
	Flocculant	46 g/t	350 t
	Coagulant	22 g/t	170 t
Consumables	Grinding media (70 mm steel balls)	0.350 kg/t	2,525 t
	Grinding media (30 mm steel balls)	0.450 kg/t	3,247 t
	Grinding media (3 mm Ceramic)	20 g/kWh	92 t

17.2 Kakula 2020 FS Processing Production Schedule

The processing production schedule is shown in Table 17.3.

The Life of mine processing schedule is shown graphically in Figure 17.2 and in Figure 17.3.

Figure 17.2 Kakula 2020 FS Processing Schedule

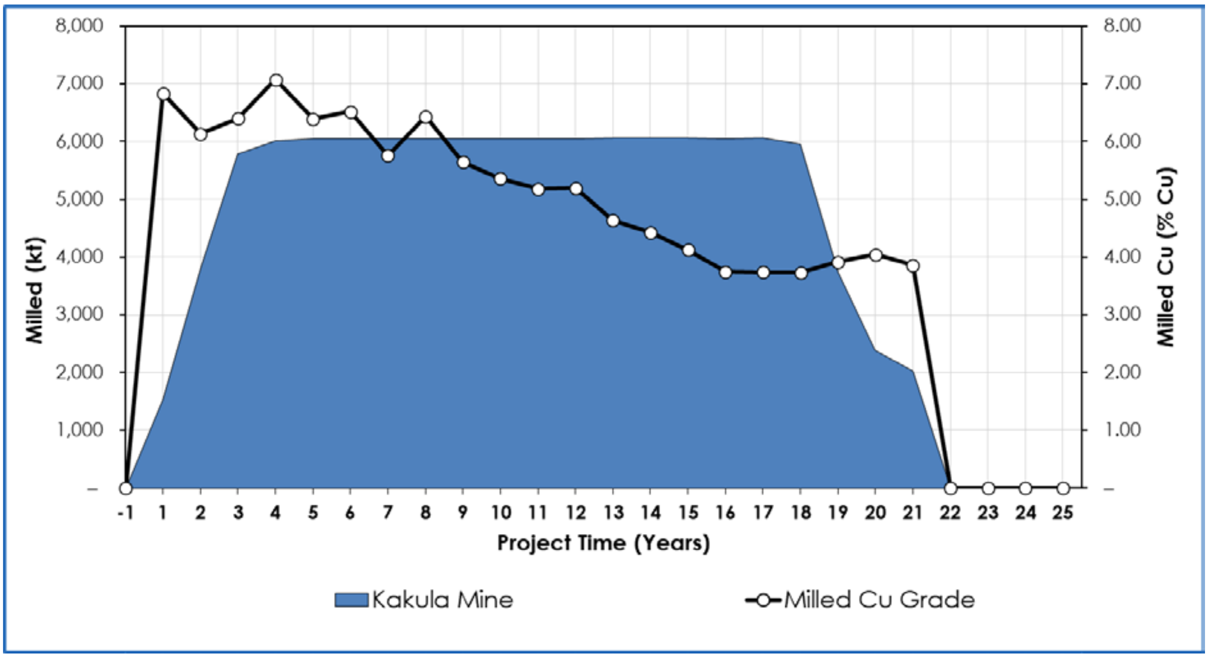


Figure 17.3 Kakula 2020 FS Processing Schedule – Concentrate Produced

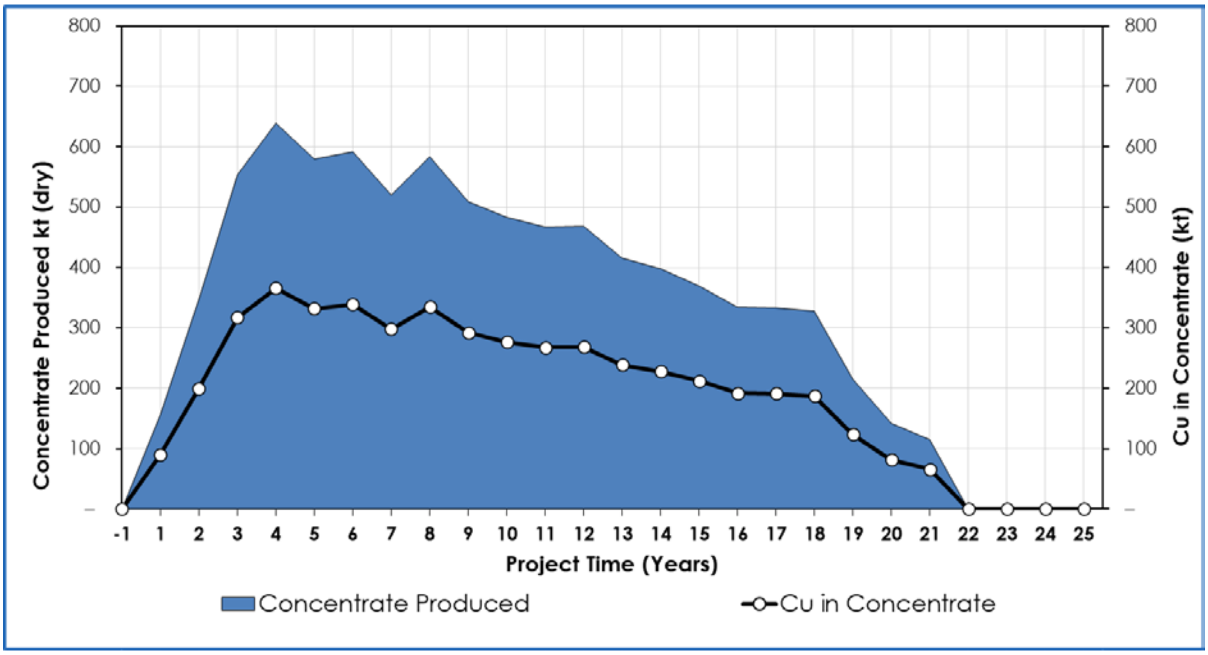


Table 17.3 Processing Production Schedule

Description	Units	Totals	Project Time (Years)														
			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Quantity Milled	kt	109,975	1,536	3,800	5,780	6,012	6,059	6,054	6,052	6,055	6,052	6,050	6,056	6,060	6,061	6,061	6,068
Cu Feed Grade	% Cu	5.22	6.83	6.13	6.40	7.08	6.39	6.52	5.76	6.44	5.65	5.36	5.18	5.20	4.63	4.43	4.13
Copper Conc. Produced	kt (dry)	8,542	153	351	547	639	579	591	520	584	509	483	467	469	416	398	370
Copper Concentrate Recovery	%	85.23	84.97	85.70	84.94	86.07	85.79	85.85	85.52	85.81	85.46	85.33	85.24	85.25	84.94	84.83	84.64
Copper 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	57.32	57.32	57.32
Total Recovered Copper Production	Mlb	10,795	193	443	692	807	732	747	657	737	644	610	590	592	526	502	467
Total Recovered Copper Production	kt	4,897	88	201	314	366	332	339	298	334	292	277	268	269	239	228	212
Description	**Units**	**Project Time (Years)**															
		16	**17**	**18**	**19**	**20**	**21**	**22**	**23**	**24**	**25**	**26**	**27**	**28**	**29**	**30**	**31**
Quantity Milled	kt	6,058	6,064	5,957	3,739	2,377	2,023	–	–	–	–	–	–	–	–	–	–
Cu Feed Grade	% Cu	3.75	3.74	3.73	3.91	4.04	3.86	–	–	–	–	–	–	–	–	–	–
Copper Conc. Produced	kt (dry)	334	334	327	216	142	115	–	–	–	–	–	–	–	–	–	–
Copper Concentrate Recovery	%	84.39	84.38	84.37	84.50	84.59	84.47	–	–	–	–	–	–	–	–	–	–
Copper Concentrate Grade	% Cu	57.32	57.32	57.32	57.32	57.32	57.32	–	–	–	–	–	–	–	–	–	–
Total Recovered Copper Production	Mlb	422	422	413	273	179	145	–	–	–	–	–	–	–	–	–	–
Total Recovered Copper Production	kt	192	191	187	124	81	66	–	–	–	–	–	–	–	–	–	–

17.3 Kakula-Kansoko 2020 PFS Processing Production Schedule

The processing production schedule is shown in Table 17.4.

The Life of mine processing schedule is shown graphically in Figure 17.4 and in Figure 17.5.

Figure 17.4 Kakula-Kansoko 2020 PFS Processing Schedule

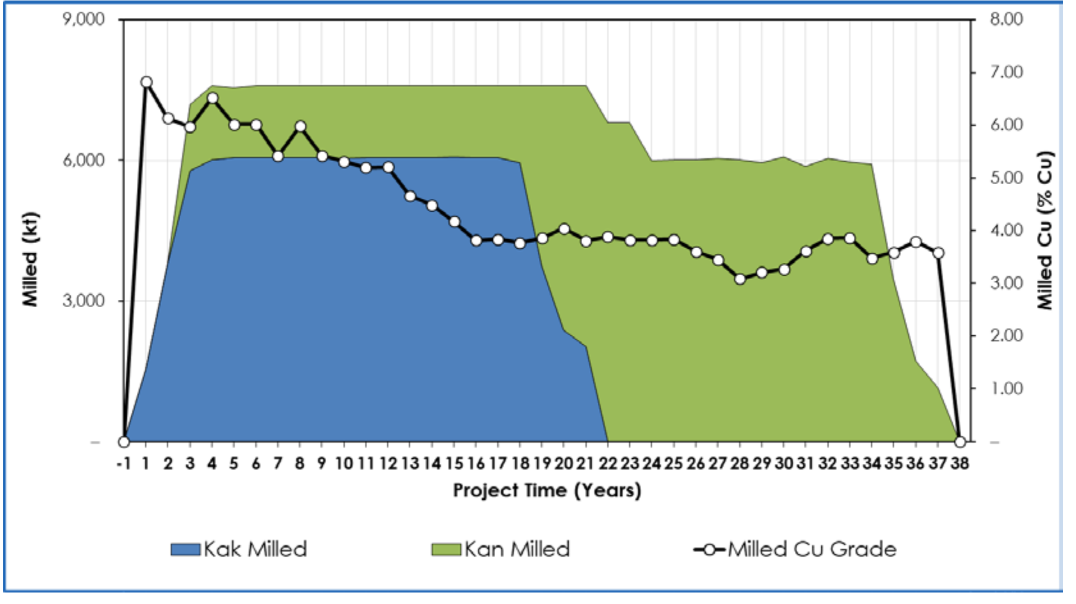


Figure 17.5 Kakula-Kansoko 2020 PFS Processing Schedule – Concentrate Produced

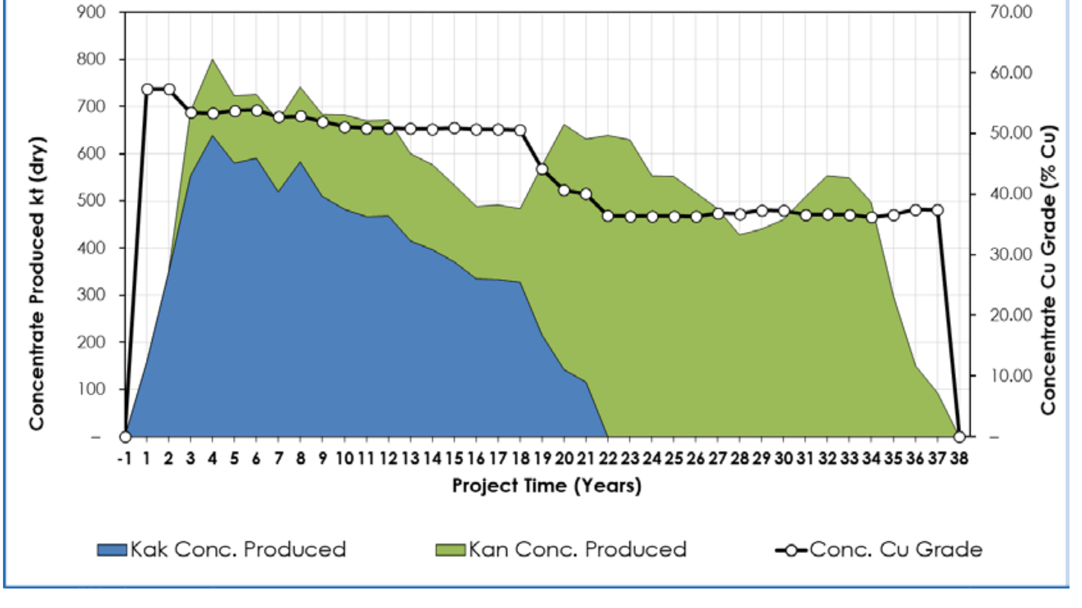


Table 17.4 Kakula-Kansoko 2020 PFS Processing Production Schedule

Description	Units	Totals	Project Time (Years)														
			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Quantity Milled	kt	235,157	1,536	3,800	7,200	7,600	7,544	7,600	7,600	7,600	7,600	7,600	7,600	7,600	7,600	7,600	7,600
Cu Feed Grade	% Cu	4.47	6.83	6.13	5.97	6.52	6.02	6.02	5.42	5.98	5.43	5.32	5.20	5.21	4.67	4.48	4.18
Copper Conc. Produced	kt (dry)	19,956	157	348	691	801	724	727	670	742	684	682	670	671	600	577	534
Copper Concentrate Recovery	%	86.32	85.97	85.69	85.90	86.16	85.71	85.62	85.84	86.23	86.15	86.29	86.28	86.29	85.98	85.88	85.62
Copper Concentrate Grade	% Cu	45.50	57.32	57.32	53.45	53.35	53.75	53.91	52.75	52.87	51.95	51.13	50.87	50.89	50.80	50.75	50.92
Total Recovered Copper Production	Mlb	20,017	199	440	814	942	858	864	779	864	784	769	751	753	672	645	599
Total Recovered Copper Production	kt	9,080	90	200	369	427	389	392	354	392	355	349	341	342	305	293	272

Description	Units	Project Time (Years)															
		16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31
Quantity Milled	kt	7,600	7,600	7,600	7,600	7,600	7,600	6,805	6,811	6,001	6,007	6,016	6,048	6,016	5,948	6,076	5,875
Cu Feed Grade	% Cu	3.82	3.83	3.77	3.86	4.05	3.81	3.89	3.82	3.82	3.84	3.61	3.45	3.09	3.21	3.27	3.62
Copper Conc. Produced	kt (dry)	489	491	484	575	662	631	639	630	554	552	519	483	427	440	460	509
Copper Concentrate Recovery	%	85.40	85.41	85.37	86.57	87.59	87.39	88.04	88.02	87.80	87.04	86.90	85.44	84.67	85.97	86.27	87.66
Copper Concentrate Grade	% Cu	50.70	50.70	50.57	44.15	40.67	40.10	36.47	36.41	36.36	36.40	36.35	36.90	36.78	37.37	37.32	36.64
Total Recovered Copper Production	Mlb	546	549	539	559	594	558	514	505	444	443	416	393	347	362	378	411
Total Recovered Copper Production	kt	248	249	245	254	269	253	233	229	201	201	189	178	157	164	172	186

Description	Units	Project Time (Years)															
		32	33	34	35	36	37	38	39	40	41	42	43	44	45	46	47
Quantity Milled	kt	6,041	5,972	5,922	3,463	1,719	1,159	–	–	–	–	–	–	–	–	–	–
Cu Feed Grade	% Cu	3.85	3.86	3.48	3.59	3.80	3.58	–	–	–	–	–	–	–	–	–	–
Copper Conc. Produced	kt (dry)	553	549	497	296	149	93	–	–	–	–	–	–	–	–	–	–
Copper Concentrate Recovery	%	87.34	87.19	87.42	87.27	85.79	83.69	–	–	–	–	–	–	–	–	–	–
Copper Concentrate Grade	% Cu	36.72	36.65	36.23	36.63	37.49	37.39	–	–	–	–	–	–	–	–	–	–
Total Recovered Copper Production	Mlb	448	443	397	239	123	77	–	–	–	–	–	–	–	–	–	–
Total Recovered Copper Production	kt	203	201	180	108	56	35	–	–	–	–	–	–	–	–	–	–

17.4 Comments on Section 17

17.4.1 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 a 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 copper sulfide 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 INFRASTRUCTURE

18.1 Kakula FS Site Infrastructure

18.1.1 Introduction

Kakula surface infrastructure comprises three main areas:

- Mining surface infrastructure – All supporting infrastructure located in the decline portal areas, such as roads, buildings, dams, services, bulk earthworks, surface ventilation fans, cooling, and LV electrical reticulation.
- Process plant infrastructure – All supporting infrastructure located in the process plant area.
- General site infrastructure – General infrastructure required to support processing and mining, such as bulk supply infrastructure (including water and power), waste management, general offices, buildings, tailings storage facility (TSF), and main access roads.

DRA was responsible for the overall scope development of the surface infrastructure, while the following companies provided input to applicable facility designs:

- Kamoa Copper SA – Project team and other consultants.
- Golder Associates Africa (Golder) – Water and waste studies and surface geotechnical studies.
- Medyas consulting and Paterson & Cooke – Backfill.
- Knight Piésold – Surface geotechnical study.
- Epoch Resources – Tailings facility design.
- R&H rail – Rail engineering and logistics study.
- Stucky – Bulk power supply infrastructure.

The surface infrastructure work by DRA Projects is reported in:

- DRA Projects, 2020. Kamoa-Kakula Project Section 05 Surface Infrastructure. DRA Projects (PTY) Ltd Report No. JCDEBR0571-PM-REP-0979 prepared for Kamoa Copper SA, dated 14 May 2020.

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 IDP20 Site Plan

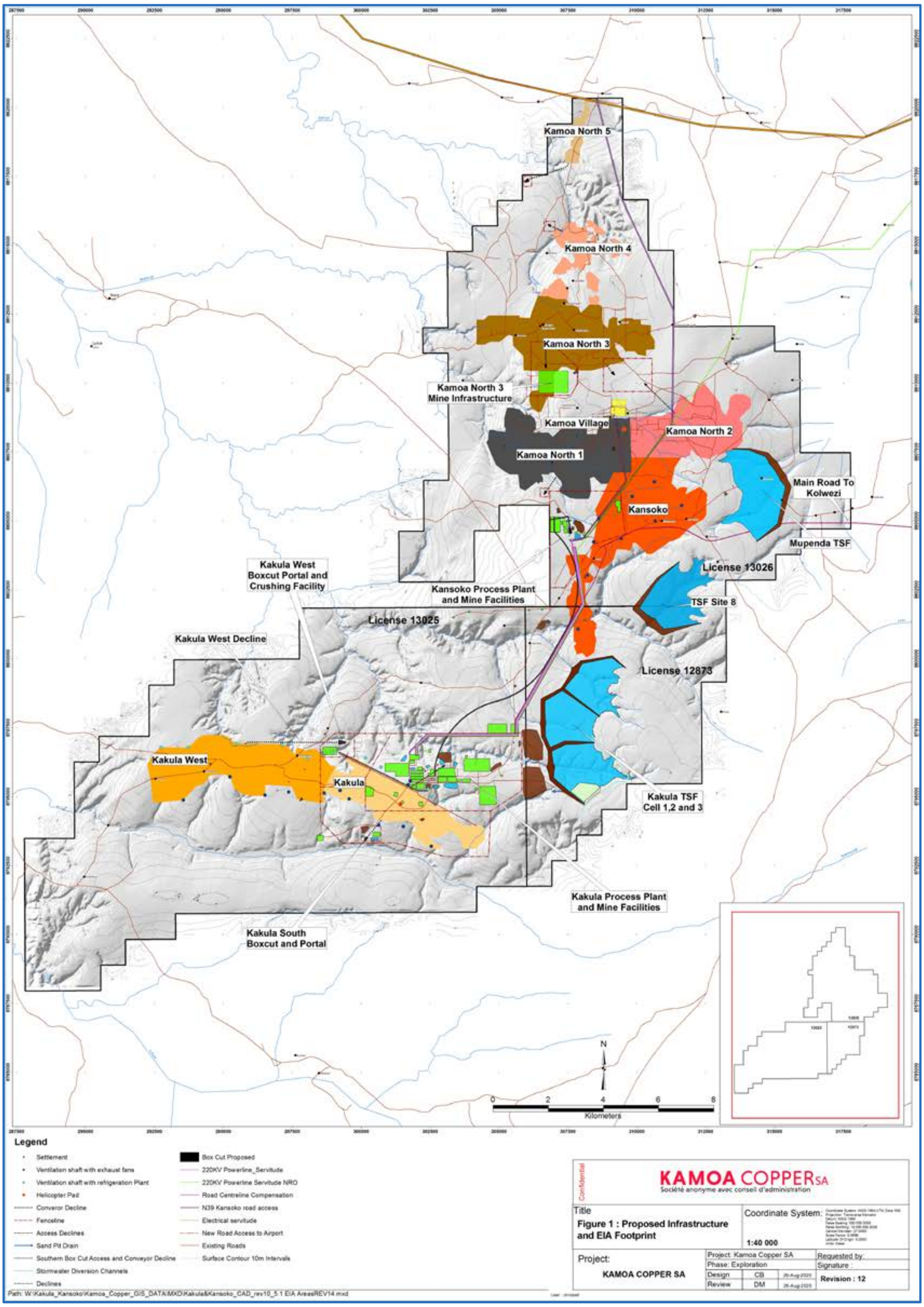


Figure by Kamoa Copper SA, 2020.

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 incorporated the philosophies agreed upon with the client. Layouts for the mining area, process plant, tailings facility and various general infrastructure were completed keeping in mind each area's needs, as well as their relation to one another. All layout design work was undertaken in close co-operation with the Kamoa-Kakula Project owner's team using the recently conducted LIDAR (light detection and ranging) survey.

The site layout was designed to allow for significant future expansion of the processing facilities, including an additional two concentrator streams and smelter, as well as overland conveyor servitudes to feed ore from other Kamoa mines to the Kakula processing facility.

18.1.4 Electrical, Control and Instrumentation Design

18.1.4.1 Power Supply

The bulk power supply is sourced from La Société Nationale d'Électricité (SNEL), the national power utility of the Democratic Republic of the Congo (DRC). Capacity from the national grid is reserved through a partnership project between SNEL and Ivanhoe Mines Energy DRC, a subsidiary of Kamoa Holding Ltd. Ivanhoe Mines Energy DRC will rehabilitate the turbine generators at the Mwadingusha hydropower plant in south-east DRC and restore the plant to its installed capacity of about 72 MW. In exchange for the upgrade of Mwadingusha Kamoa will be entitled to receive 100 MW of power from the grid, which is sufficient for the 6.0 Mtpa Kakula mine and concentrator. The upgrading is part of a programme planned to eventually overhaul and boost output from a total of three hydropower plants. On completion of the upgrading programme a combined total of 200 MW of long-term, clean electricity will be produced for the DRC's national grid. The upgrade of the three hydropower plants secures the electrical capacity required for future expansion phases of the project.

Operations power will be supplied to a Kamoa Copper SA 220 kV substation from the new SNEL substation, called Nouveau Répartiteur Ouest (NRO). The NRO substation will be financed by Ivanhoe Mines Energy DRC and forms part of a loan agreement. A double circuit 220 kV 450 MVA transmission line (35 km) will be installed between NRO and the Kamoa 220 kV substation. Metering will be at the take-off point at the NRO substation, as the overhead line will remain the property of Ivanhoe Mines Energy DRC. Kamoa is constructing a main 220 kV consumer substation at Kakula that will step down the voltage to 33 kV via two 80 MVA transformers. This substation is envisaged to be the main substation for future expansion phases of the Kamoa-Kakula project.

Construction power is presently being supplied to Kamoa from the SNEL network via a 120 kV line constructed for the project and mobile 18 MVA substation. This infrastructure will be used as a back-up power supply after commissioning of the 220 kV system.

A MV generator farm (16 x 1,800 MW MV continuous rated units) will generate backup power to all facilities at Kakula. It will be possible to select any of the electrical equipment to operate on back-up power during a SNEL outage, up to a maximum load of 18 MVA. This typically includes mine de-watering and ventilation, concentrator thickeners, and tailings pumps. It is not envisaged to operate any direct production equipment on emergency power as the fuel cost could be prohibitive.

18.1.4.2 Generation

Power for the 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$4.5M 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 modernisation of Mwadingusha, is expected to be restored to its installed capacity of about 71 MW. After completion and handover, HPP Mwadingusha will supply electrical energy to the Congo National Grid as well as to the copper mining activities at the 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 plants is shown in Figure 18.2.

Figure 18.2 Power Plants Locations

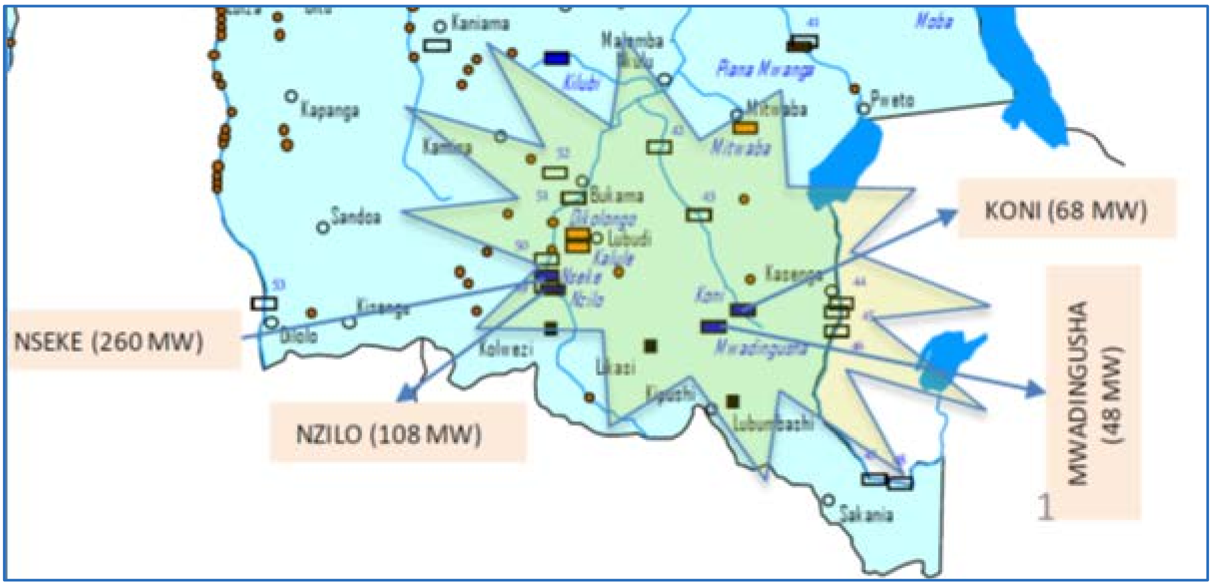


Figure by Ivanhoe, 2017.

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

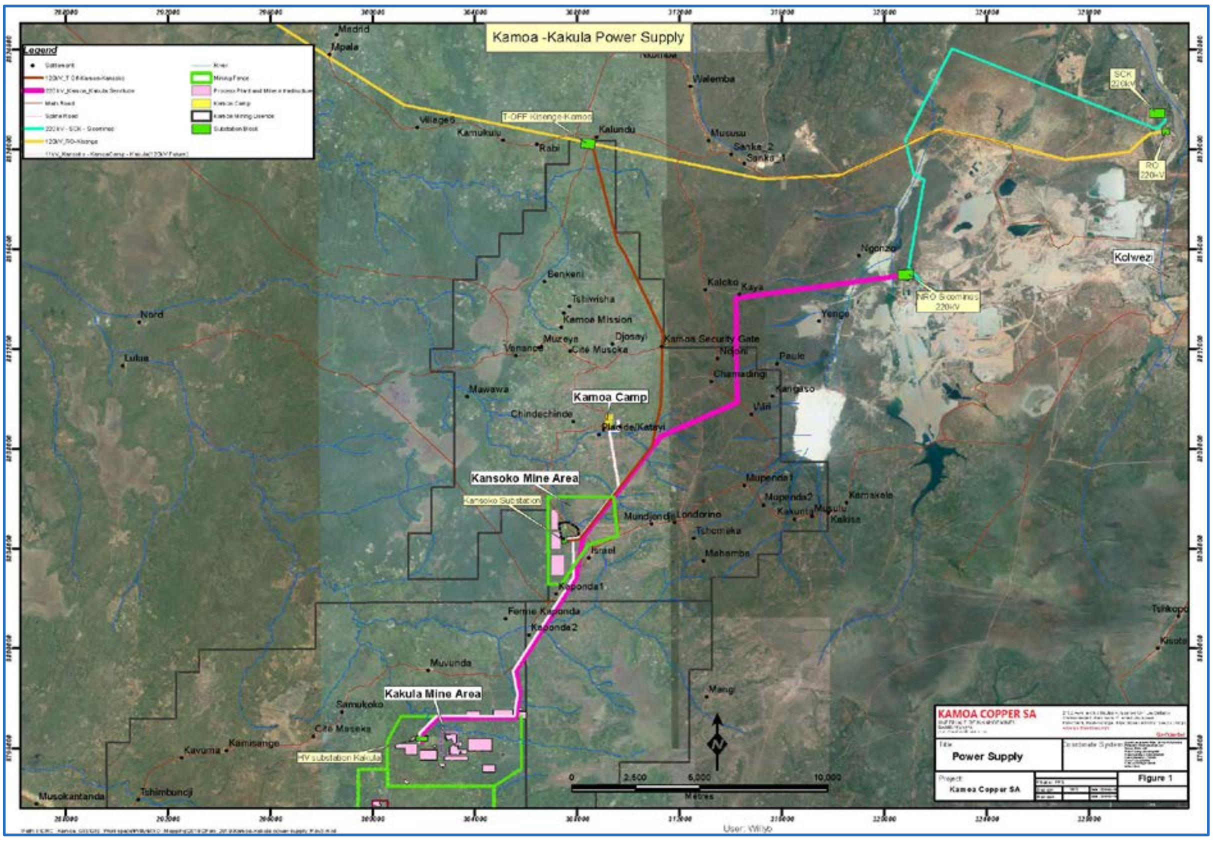
The Nzilo hydro power plant is located on the Lualaba River, approximately 30 km from the city of Kolwezi in the DRC province of Lualaba. The hydro facility was built in 1952 and comprises four turbines with an installed generation capacity of 108 MW at a gross hydrostatic head of 74.5 m. Three out of four turbines installed are currently operational but need to be renewed.

18.1.4.3 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



Three 220/33 kV 80 MVA transformers will be installed. The supply will provide N+1 redundancy on the transmission line and the transformers. KCS will be able to expand to meet all the future needs of the Kamoa project expansion, including future mines, concentrators and smelter. Construction of the 220 kV overhead line (OHL) and substation is in progress and scheduled for commissioning in December 2020. The incoming power will be monitored at the Kamoa 220/33 kV substation.

18.1.4.4 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 maximum demand (MD) for the various surface and underground proposed installations. The MD is the maximum electrical power demand in kVA, over a 15-minute period. The load estimate was calculated by generating a load list per area in MS Excel, with all the power requirements as indicated in the mechanical equipment list (MEL).

The MEL and process flow diagrams (PFDs) were used as inputs to the load lists. The mechanical power requirements are subjected to load capacity de-rating, diversity and utilisation factors to compensate for the operating conditions and to obtain a realistic value for the running power (MD). The mining and production schedules were also applied to the running loads to obtain an MD load profile in kW. See Table 18.1 for a summary of the running load by area.

Table 18.1 Kakula Notified Maximum Demand

	Running MW
Mining Crew	10.23
Ore Transport	12.94
Dewater	25.68
Primary Ventilation	13.19
Cooling	4.78
Services	5.55
Mining	72.37
Backfill Ph-1	4.59
Backfill Ph-2	3.36
Backfill	7.95
Concentrator Ph-1	21.14
Concentrator Ph-2	18.54
Concentrator	39.69
Total	120.00

18.1.4.5 Construction Power Supply

Power supply for the construction period is sourced from the existing 120 kV line. An existing 18 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 Kansoko mine and will be relocated to the planned 220 kV substation yard at the Kakula mine.

18.1.4.6 Medium Voltage Electrical Distribution

The selected voltages for the Kakula 2020 FS are as follows:

- Medium voltage systems:
  - Distribution voltage: 33 kV surface and 11 kV underground AC resistively earthed.
  - Nominal frequency: 50 Hz.
- Low voltage systems:
  - Mine developing equipment operating voltage: 1,000 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), which also caters for harmonic filtering, will be implemented at the medium voltage level to take advantage of the benefits of scale. A distributed PFC philosophy has been applied and provides greater flexibility in terms of incremental introduction as the site load increases. A total of 35 MVAr PFC has been allowed for the final position and will be determined during the execution phase

Power will be distributed from the 33 kV KCS substation to the following substations:

- 33 kV KCS substations.
- 33 kV concentrator substations.
- 33 kV backfill plant substation.
- 33 kV distribution substation – north.
- 33 kV distribution substation – south.
- 33 kV VS NE1.
- 33 kV east refrigeration plant substation.
- 33 kV VS NE3.
- 33 kV VS NE5.
- 33 kV west refrigeration plant substation.
- 33 kV VS NW2.
- 33 kV VS NW5.
- 33 kV VS SE#1.
- 33 kV VS SE#2.
- 33 kV VS SW#1.
- 33 kV VS SW#2.
- 33 kV VS SW#3.

18.1.4.7 Back-up and Standby Power

A MV generator farm (16 x 1,800 MW MV continuous rated units) will generate backup power for the planned power system. The generator supply will be for emergency loads (e.g. mine pumping) and not for production although the design allows for co-generation with the SNEL supply. Generated power will feed into the MV network during power outages. A power management system (PMS) will be commissioned to monitor all MV switchboards, synchronise the generators, switch off non-essential breakers and optimise the efficiency of the generator power plant.

18.1.4.8 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.

18.1.4.9 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/400 V and 11 kV/400 V Dyn11 ONAN transformers and mini substations. These include the numerous offices, workshops, change houses and other similar facilities. The neutral point of the 400 V 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/400 V 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.

18.1.4.10 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 operations of the concentrator plant and the backfill plant.

All electrical feeds and plant statuses are monitored and logged, and form the basis of PFC, energy management (EM) and power management (PM). Communication between all the relevant points will take place over optic fibre utilising overhead lines and existing routes. Field input/output (I/O) will be a combination of ASi-Master panels for conveyors and remote input/output (RIO) panels for other areas.

18.1.4.11 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) were requested to undertake a feasibility study for a Tailings Storage Facility (TSF) for a 6.0 Mtpa mine at the Kakula Mine for the Kamoa-Kakula Project, on behalf of Kamoa Copper SA.

The work is reported in:

- Epoch Resources, 2020. Feasibility Study for a Tailings Storage Facility at the Kakula Mine as part of the Kamoa-Kakula Project. Project Number 211-006, Report No.1 Final, dated May 2020.

18.1.5.1 Scope of Work

The scope of work for the feasibility study of the TSF was as follows:

- Stage capacity characteristic curves (area-volume-height curves) for the TSF.
- Seepage analyses for the TSF.
- Slope stability analyses of the TSF containment wall.
- A water balance of the TSF to determine typical return water volumes.
- The design of the TSF comprising:
  - A tailings impoundment that accommodates 61.2 million dry tonnes of tailings over a 25-year Life-of-mine.
  - The associated infrastructure for the TSF such as the perimeter slurry deposition pipeline, pool access wall, and storm water diversion.
- Site layout and typical drawings of the TSF.
- Estimation of the capital costs to an accuracy of ±15% and operating costs associated with the TSF to an accuracy of ±25%.
- Estimation of closure, rehabilitation, and aftercare costs to an accuracy of 35%.
- Estimation of the costs over the life of the facility.

18.1.5.2 Design Criteria

The Kakula TSF have been designed with a LOM tailings production of 61.2 Mt over 25 years. The Kakula LOM is 21 years. The particle specific gravity (SG) of the tailings ware determined to be 2.92, by Specialised Testing Laboratory (Pty) Ltd (ST Lab). The design criteria are summarised in Table 18.2.

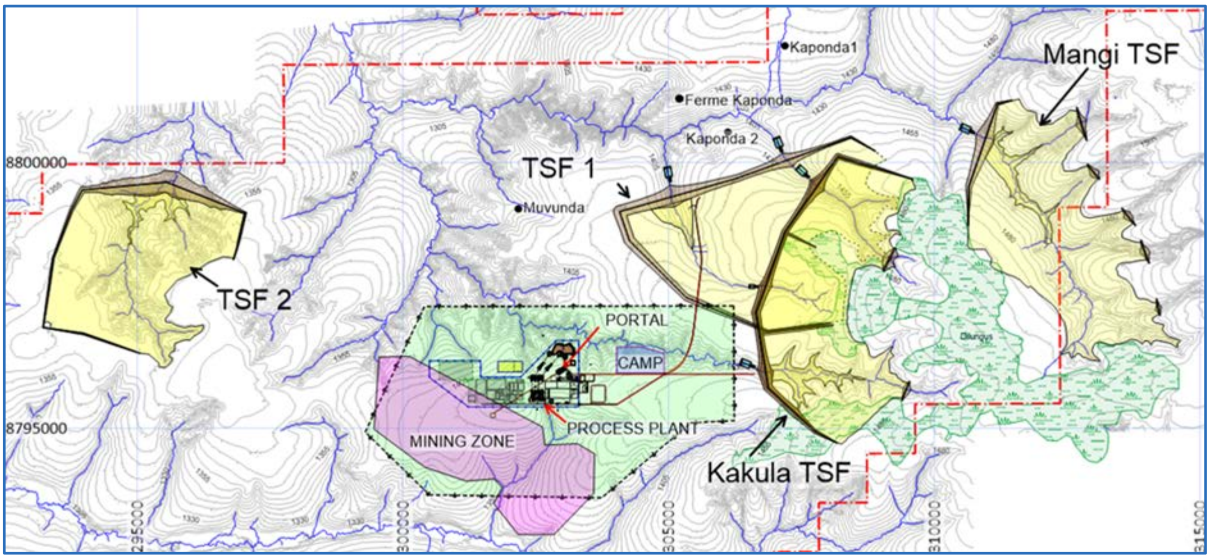
Table 18.2 Design Criteria

Item	Criteria	Value	Source
1	Ore type	Copper	Kamoa
2	Design life of facility	25 years	Kamoa
3	Average tailings production rate	5.56 Mtpa	Kamoa
4	Average tailings backfill rate	2.33 Mtpa	Kamoa
5	Average tailings deposition rate	3.23 Mtpa	Kamoa
6	Total tailings	61.2 Mt	Kamoa
7	Raise 1 storage requirement	One year of capacity (minimum)	Kamoa
8	Particle specific gravity	2.923	ST Lab
9	Average dry density	1.28 t/m^3	ST Lab/Epoch
10	Average particle size distribution	75% passing 60 µm	ST Lab
11	% solids to water ratio (by mass)	48	DRA/Kamoa
12	Delivery method	Hydraulically pumped	Kamoa
13	Geochemistry	Leachable mine waste	Golder

18.1.5.3 TSF Site Selection

A site selection study was undertaken as part of the preliminary economic assessment in 2017. The required capacity for the study was for 228 Mt, to accommodate for possible further expansion. The preferred site was found to be the Kakula TSF site, shown in Figure 18.4.

Figure 18.4 Sites Identified for the TSF



18.1.5.4 Geotechnical Investigation

A geotechnical investigation of the TSF site was undertaken by Knight Piésold. This included profiling of test pits and boreholes, sampling of soils, and laboratory test work of the samples.

The soils encountered at the TSF are characterised by transported soil of mixed origin, but mainly of aeolian provenance, along the upper slopes of the valley, and colluvium and hillwash along the valley floor. The aeolian comprises a silty to sandy, very loose to loose material, with rapid seepage and collapsing sidewalls.

Groundwater seepage was encountered in several test pits during the investigation at varying depths below ground level. The geotechnical investigation was carried out between 24 November 2018 and 22 April 2019, which coincides with the regional wet season. Furthermore, the site is positioned within a valley and overlays the lower reaches of several drainage lines. It would, therefore, be expected that a higher water table is experienced across the test site.

Knight Piésold noted that the properties of most of the soils in the vicinity of the TSF constitute a poor construction material due to:

- Poor compaction – Committee of Land Transport Officials (COLTO) classification of mostly G8 and G9 material with poor Californian bearing ratio (CBR) results.
- Potential dispersiveness.
- Mostly silty transported soils, with high plasticity.

18.1.5.5 Seepage ad Stability Assessment

The stability of the TSF was assessed under various seepage conditions. The results show that the TSF containment wall is stable, with a factor of safety well above the minimum of 1.5. Abnormal operating conditions, such as a damaged liner or the absence of drains, emphasises the contribution these design elements make towards providing stability within the various phases of the TSF. It has been shown that without these elements the factor of safety would reduce to below the minimum required levels.

18.1.5.6 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 tailings will be discharged from the top of the containment wall crest creating a beach, resulting in a supernatant pool developing 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.

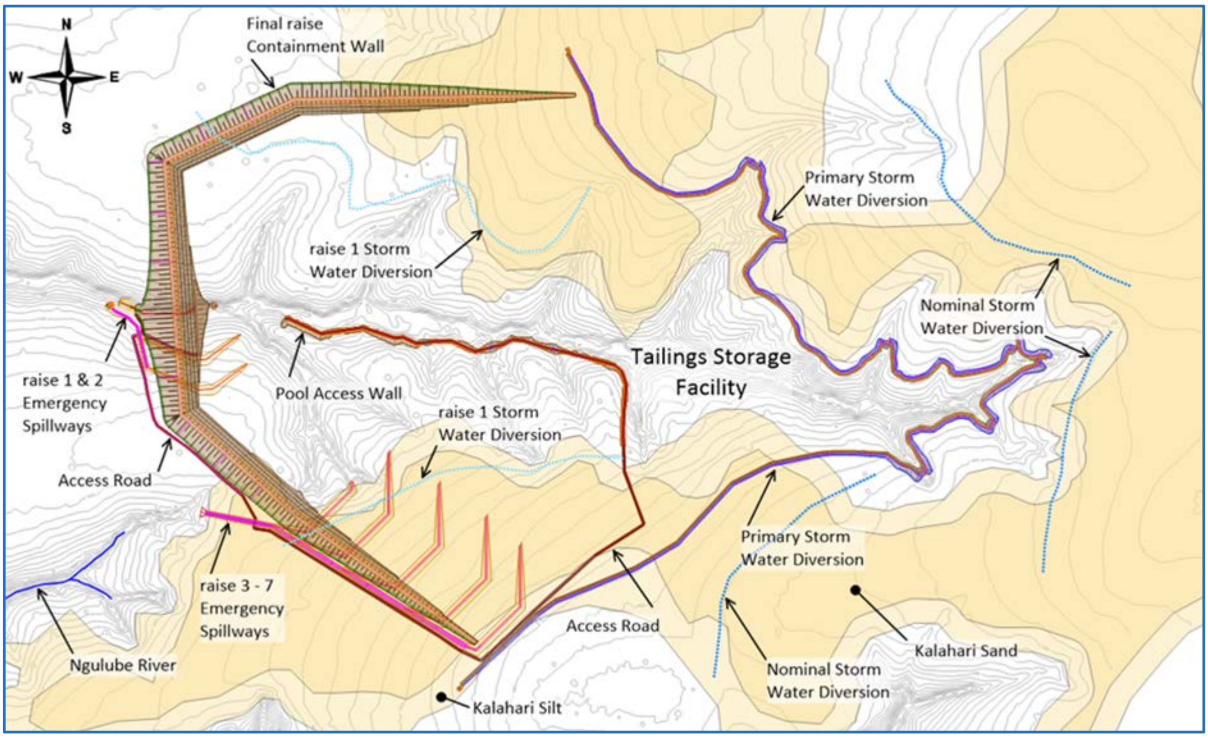
For the selected depositional methodology, tailings are deposited into the TSF basin via an open-ended pipeline located on the inner crest of the containment wall. During commissioning, deposition of the tailings behind the containment wall is directed to the base of the inner toe of the containment wall by flexible hoses. Deposition during this stage is to be carefully controlled, monitored, and intensely managed to ensure that the wall is not eroded by the tailings stream.

18.1.5.7 Key Design Features

The layout of the TSF is shown in Figure 18.5 and the key design features of the facility are as follows:

- Full containment, downstream construction method, with open-ended deposition.
- An engineered, earth-fill containment wall.
- Pool access wall.
- A floating decant system.
- Curtain and blanket drain seepage collection system inside the containment wall (to reduce the phreatic level within the wall).
- An HDPE liner and associated seepage cut-off drains.
- Storm water diversion trenches.

Figure 18.5 TSF Layout



18.1.5.8 Risk Identification

The major risks associated with the TSF are as follows:

- Due to the large catchment upstream of the TSF, it is recommended to schedule construction during the dry months, as this will result in less delays and easier working conditions. The programme set up by the contactor must take this into account.
- The preferred site lies upstream of the proposed infrastructure and the accommodation camp and these facilities may be affected by a breach of the containment wall. An attempt has been made to move this infrastructure to higher ground, however a risk still exists that it may be affected by a tailings flow slide.
- Availability of construction materials could pose a significant threat to the cost and time of construction. It must be ensured that materials will be available on-site before and as construction commences.
- Suitable borrow areas close to the TSF site cannot be identified and material must be sourced from further away resulting in delays and increased costs.
- The liner requirements for the basin of the TSF have not been finalised. Approval must be obtained from authorities. Lining the entire basin would result in significant increased capital cost.

- The TSF failing and causing a flow slide is a key risk. This must be managed through an intense QA/QC system, construction management and supervision during the construction of the facility, and competent operational management, so as to reduce the risk of failure. More specific issues and mitigation measures are identified including:
  - Compaction of the containment wall must be performed according to specification to ensure no ratholing or piping occurs as these could lead to failure.
  - The entire perimeter of the TSF must be inspected on a daily basis to ensure any defects are noted as early as possible. Such as: sloughing, slips, ratholing, or seepage.
- The water levels on the TSF must be monitored to ensure that sufficient water is pumped off the TSF to provide sufficient storage for the design storm event. Failure to provide sufficient storage may lead to water overtopping the containment wall. Standby pumps and generators are required to ensure sufficient pumping capacity.
- Tailings elevation inside the TSF must be monitored to ensure that the construction of the wall raises are scheduled correctly to ensure tailings freeboard in maintained and capacity is available.

18.1.5.9 Conclusions

The following conclusions were deduced from the studies documented in this report:

- From the seepage and slope stability analysis for the TSF, it was found that based on the parameters determined from the test work and the geometry of the TSF, the facility should be stable, with a factor of safety above 1.5.
- The TSF has been designed to store a total of 61 million dry tonnes of tailings over a period of 25 years and comprises:
  - A TSF, with a footprint area of 420 Ha and a maximum height of 50 m from the lowest contour.
  - Associated Infrastructure such as storm water diversion and curtain drains.
- The water balance model indicated that on average 8,748 m^3 may be returned to the process plant circuit per day.

18.1.5.10 Recommendations

The following recommendations are provided for the detailed design phase of the project:

- Confirm design criteria.
- Final tailings sample be tested (if mill grind or other physical characteristics differ from the sample tested).
- Identification of appropriate borrow area for later raises.
- Confirm with the DRC authorities the liner requirements for the basin of the TSF.

- The next phase of the study should aim to further improve capital and operating cost estimate, where possible, by:
  - Undertaking a tender enquiry on the detailed design to acquire final construction rates.
  - Further optimisation of earthworks and civil-works, where possible.
  - Finalising the responsibilities of the operator by incorporating input from all parties (contractor, client, and consultants).
- Confirmation of survey data accuracy. It is recommended to undertake survey points of the site to confirm elevation (particularly, in the area where sand mining will change the topography).

18.1.6 Site Waste Management

The waste management system was designed to cater for 3,000 people at the peak of production. A non-hazardous landfill site will be constructed at Kakula that will handle the general waste generated by the mine and domestic waste generated by the mine personnel. Hazardous waste storage facilities have also been included to store wastes such as chemicals until there is a suitable disposal site available in Kolwezi. Old oil and batteries will be recycled or re-used by various local businesses. Old tyres will be placed in mined-out drifts before backfilling.

18.1.7 Earthworks

The ground conditions at Kakula are very well suited for constructing the surface facilities; no soft ground or clays and no blasting required. The topography is also well-suited; not too steep, that would require extensive terracing and not too flat, that would present drainage problems. All waste rock from mining development will be used for constructing terraces and roads.

18.1.7.1 Terraces

Knight Piésold was appointed by Kamoa Copper SA (Kamoa) to provide geotechnical engineering consulting services for the project. Knight Piésold conducted observations from the field component of the geotechnical investigation conducted at the proposed site for the processing plant and the TSF, as well as an analysis and interpretation of the laboratory test results of the soils sampled.

Geotechnical conditions encountered at the site are classified and foundation requirements are outlined for planning of surface mine infrastructure. It is anticipated that additional drilling will need to be conducted to confirm the rock properties for heavy structure foundation design. Table 18.3 presents the foundation recommendations based on the field findings.

Table 18.3 Foundation Recommendations Based on Field Findings

<table>
<tr><th rowspan="2">Category</th><th rowspan="2">Possible Infrastructure</th><th colspan="2">Foundation Recommendations</th></tr>
<tr><th>Zone 1 (Shallow hard rock)</th><th>Zone 2 (Low to medium stiffness)</th></tr>
<tr><td>Light structures (50–100 kPa)</td><td>Single-storey buildings</td><td>Conventional strip and pad footing on medium dense colluvium / residual soils or siltstone rock</td><td>Engineered soil mattress.
Excavate and recompact granular soils to ±1.5 m below foundation level, or stiffened raft foundation – compatible with residual ground movements and top structure flexibility. Strict QA/QC controls.</td></tr>
<tr><td>Medium and/or lightly dynamically loaded structures (±250 kPa)</td><td>Conveyors and compressors, etc. (without significant moment about toe pressures)</td><td>Deep foundations into siltstone rock</td><td>Conventional deep founding into proven medium dense to dense colluvium / residual soils. Alternatively, engineered soil mattress as above (full depth) and stiffened raft foundation.
Strict QA/QC controls.</td></tr>
<tr><td>Heavy structures and/or settlement sensitive elements (±750 kPa)</td><td>Silos, crushing plants and mills</td><td colspan="2" rowspan="2">Pile foundations, typically from an average depth of 10 m on medium hard siltstone rocks or better. Depths to be confirmed from the proposed detailed investigation, including drilling.</td></tr>
<tr><td>Ultra-heavy structures (± 3,000 kPa)</td><td>Very large crushing plants with large dynamic loads</td></tr>
</table>

The above recommendations are generic and preliminary and must be expanded and confirmed in site-specific directives by the Geotechnical Engineer once detailed structural and foundation details are available.

Low-Specification Terraces will be used in laydown areas and temporary terraces required for the construction phase of the project, as well as typical buildings and stores with a bearing capacity of 50–100 kPa. The following areas will require low-specification terracing:

- Workshops.
- Offices.
- Stores.
- Change house.
- Gate houses.
- Waste facilities.

Medium-Specification Terraces are terraces that cater for structures where the design requires a bearing capacity of 150 kPa.

High-Specification Terraces are required to cater for large, high and vibrating structures where the design requires a bearing capacity of 250 kPa. The following areas will require high-specification terracing:

- 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

External roads: include the construction of the main access road initially and the Katonoto export road later in mine life. The main access road connects the mine with the airport and main incoming Kolwezi routes directly, improving access to site for material and personnel transport.

The Katonoto export road will connect with the westbound route to Angola and will be used as a concentrate export route.

Each of these roads will be constructed as gravel roads initially. The option to asphalt the roads later during LOM remains an option.

The internal road design philosophy is that delivery vehicles, light-duty vehicles (LDVs) and concentrator trucks will remain on separate roads to the required delivery points, working and parking areas. Internal roads will be reserved for equipment delivery from stores to the applicable work areas. The truck parking area at the main gate has been sized to accommodate 300 trucks. The internal roads and parking will consider the traffic flow inside the mine area. Security gates separate areas in order to control access, without reducing serviceability and production. The surface finish of all roads will be gravel. Layer works have been designed for different applications, traffic and road uses in the mine area.

18.1.7.3 Storm Water Management

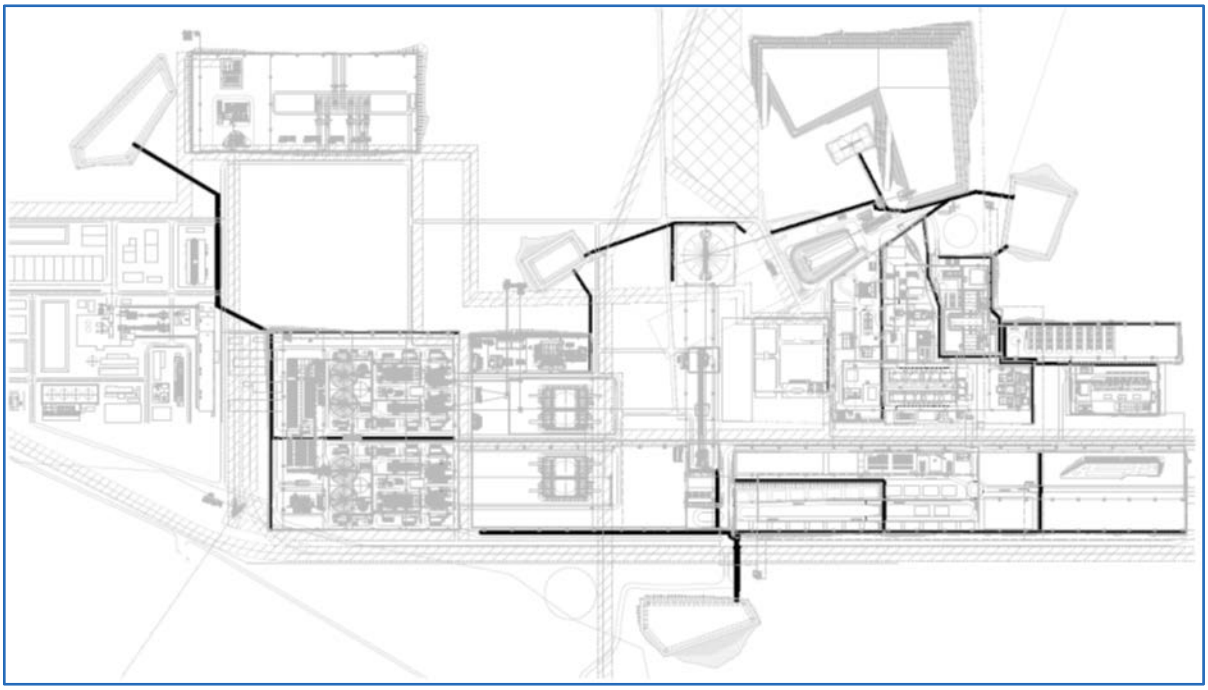
Golder Associates Africa modelled the run-off and made recommendations, based on the following:

- 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 18.6 shows the layout of the primary collection drains. Additional secondary drains have been positioned to channel all run-off water.

Figure 18.6 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. All stormwater management is planned to commence 18–24 months after the start of construction.

18.1.7.4 Water Storage Facilities

Different types of dams have been allowed for in accordance with their varied requirements. Table 18.4 indicates the water storage facilities allowed for, and their associated capacities.

Table 18.4 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	41,537,828 m^3
Return Water Sump	Earth dam	1,400 m^3
Process Water Dam	HDPE lined earth dam	15 ML
Mine Service Water Tank	Mild steel	5,000 m^3
VD2 Mine Service Water Tank	Mild steel	500 m^3
Process Water Tank #1/#2	Concrete	2 x 3,000 m^3
Concentrator Filtered Water Tank	Concrete	1,000 m^3

18.1.7.5 Material Storage Stockpiles

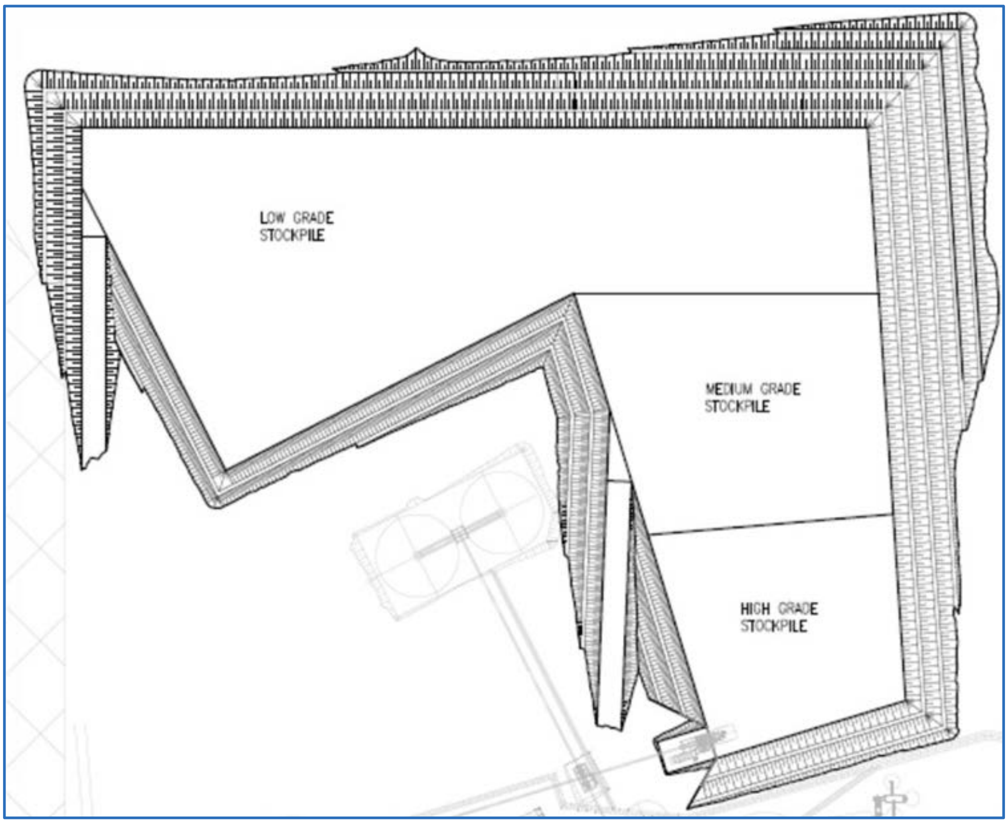
Both ore and waste rock stockpiling encompasses the stockpiling of all commodities generated during both construction, and operational phases. These commodities include:

- Low-grade (LG) ore.
- High-grade (HG) ore.
- Waste rock uncrushed and crushed.
- 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 18.7 below for and illustration of the bulk ore stockpiles.

Figure 18.7 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 South African National Standards, 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 are designed to enable collection from various facility points for outing through a 100–250 nominal bore (NB). Depending on the flow, a polypropylene gravity-fed buried pipe system, that is connected to a concrete-lined sump at the sewer plant, will be used. There will be two sewer treatment plants. One will be located at the Kakula village and the other will be used for the contractor's camp and mining facilities (including the change house). Each plant is designed for 1,600 people. See Figure 18.8 for a schematic of the treatment plant, including all the required treatment components.

Figure 18.8 Three-Stage Phoredox Process

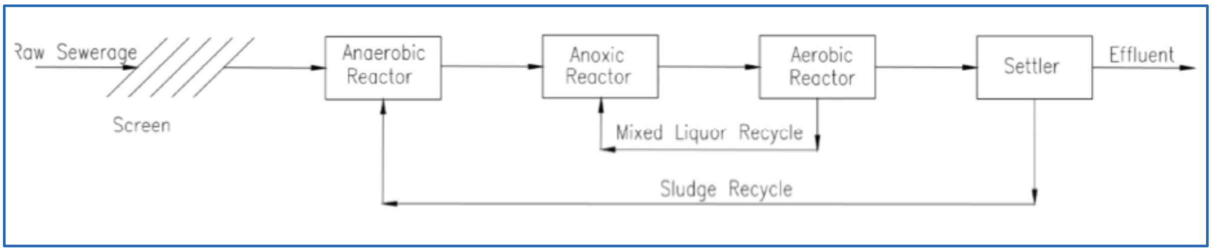


Figure By: DRA, 2020.

The digested sludge in the septic tanks can be emptied by tanker truck once a year (or as required) and discharged to landfill. The plant will be connected to a gravity sewer pipe.

Each SAFF wastewater treatment system has a standard design to meet the requirements for sewage feed as specified. The expected final (treated) water quality complies with the RSA General Specification - Revision of general authorisation in terms of section 39 of the National Water Act 1998 (Act no 36 of 1998). Additional sand filters and activated carbon filters can be offered with the package if required.

The effluent collected in each of the sumps is pumped to a sewage treatment plant. Manholes, in accordance with SANS 1200L, have been allowed for every 50 m in order to clean and maintain the pipelines. The plant will be situated in the integrated waste management area.

18.1.7.9 Weighbridge

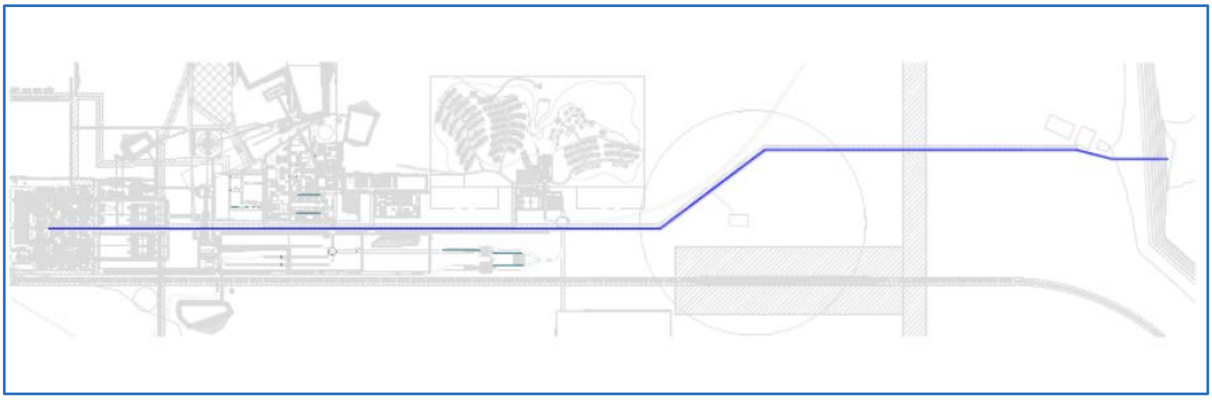
Three weighbridges have been included in the design. Two are allocated after the main gate before the bonded yard. One is in the plant area specifically to measure concentrate and materials.

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 3.8 Mtpa concentrator's, two HDPE lines in Phase 1(3.8 Mtpa) and a third in Phase 2 (7.6 Mtpa). 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 18.9 below.

Figure 18.9 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 18.5.

Table 18.5 Security and Access Control Infrastructure – Overview per Area

	Kakula Camp	Process Plant	Mining Area	Weighbridges	Security Main Control Room
Turnstiles	✓	✓	✓	×	×
Fencing	✓	✓	✓	×	×
Boom Gates	✓	✓	✓	×	×
RFID Tags or Biometric Readers	✓	✓	✓	✓	✓
Day/Night Cameras	✓	✓	✓	×	×
CCTV	✓	✓	✓	✓	✓
Workstation	✓	✓	✓	✓	✓
UPS	×	✓	✓	✓	✓
Office Alarm System	×	✓	✓	×	✓
Computerised Access Control	×	✓	✓	✓	×

18.1.9 Logistics

R&H Rail (Pty) Ltd (2020) completed a rail and logistics study for Kamoa Copper SA. A phased logistics solution is proposed for transport of Kakula concentrate during operations. During the initial production years, the North-South corridor between southern DRC and Durban or Richards Bay in South Africa is considered the most attractive and reliable export corridor. Bagged concentrate product will be packed on-site and transported by truck to Ndola in Zambia, where it will be loaded onto trains and transported to the Durban or Richards Bay port.

Later during project life an export road (Katonoto export road) will be constructed that will allow export trucking to be done via a main export road connecting to the west to the Angolan rail / road network and the ports of Lobito or Luanda.

On-site infrastructure is planned to support the import / export logistics including a large secured truck park area, bonded yard for customs clearing, weigh bridges, and product sampling systems.

A new 30 km access road has been constructed by Kamoa from Kolwezi to Kakula Mine. This road has been designed for high traffic volumes and for relatively high speeds (90 km/h). It is a gravel road, but the foundations are suitable for later asphalting of the road. This road will provide a direct link to the main road to Lubumbashi and other industries in the region including a copper smelter and cement plants. This road will also reduce commuting times between Kolwezi and Kakula to 30 minutes, making it feasible for much of Kakula's labour to commute daily from Kolwezi.

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.

The western rail corridor to Lobito and the North–South corridor through to Zambia is shown geographically in Figure 18.10. The North–South Corridor is shown in diagrammatically in Figure 18.11.

Figure 18.10 Kamoa to Lobito Rail System

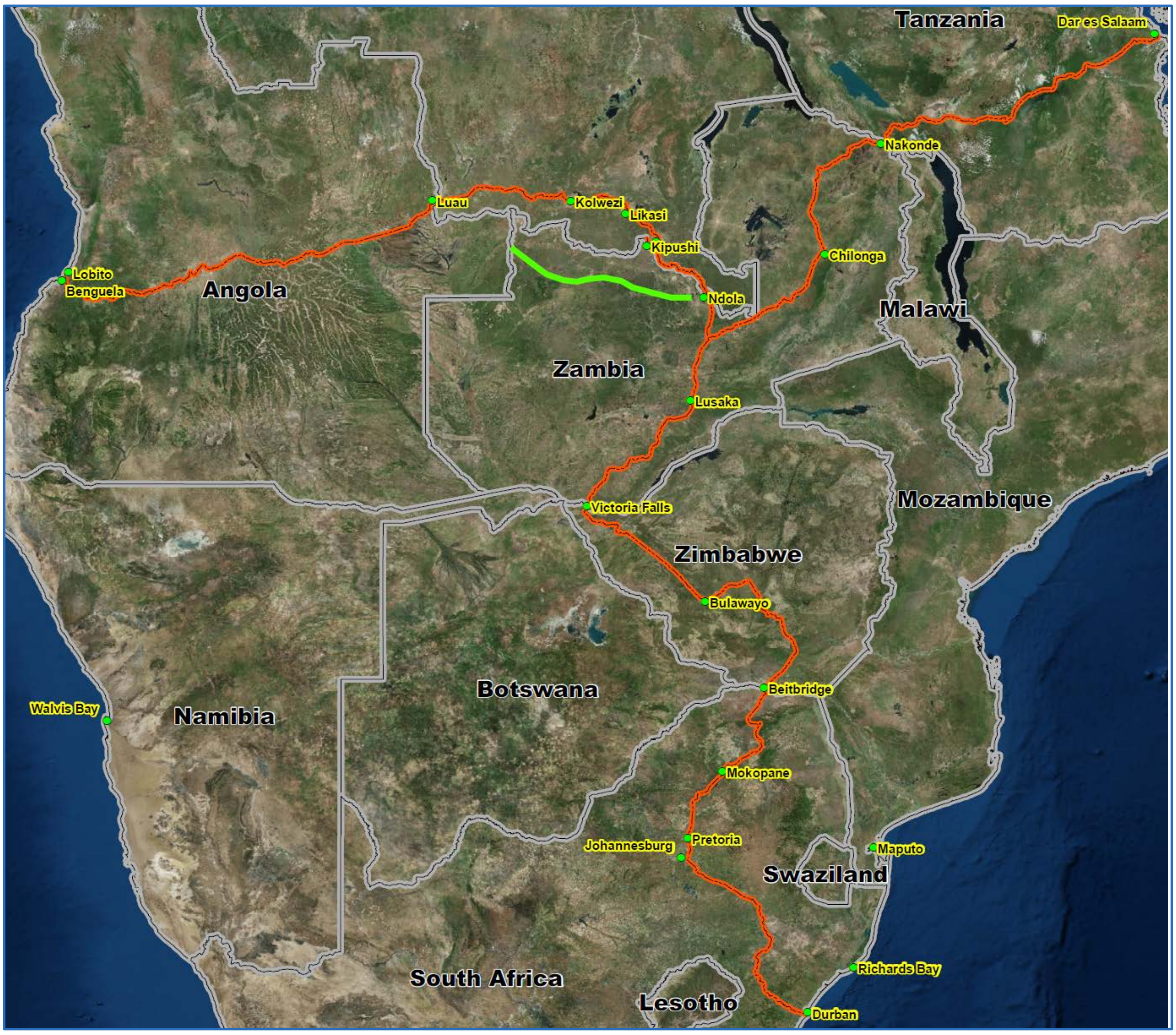


Figure by Grindrod Limited, 2017.

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 18.11 DRC to South Africa North–South Rail Corridor

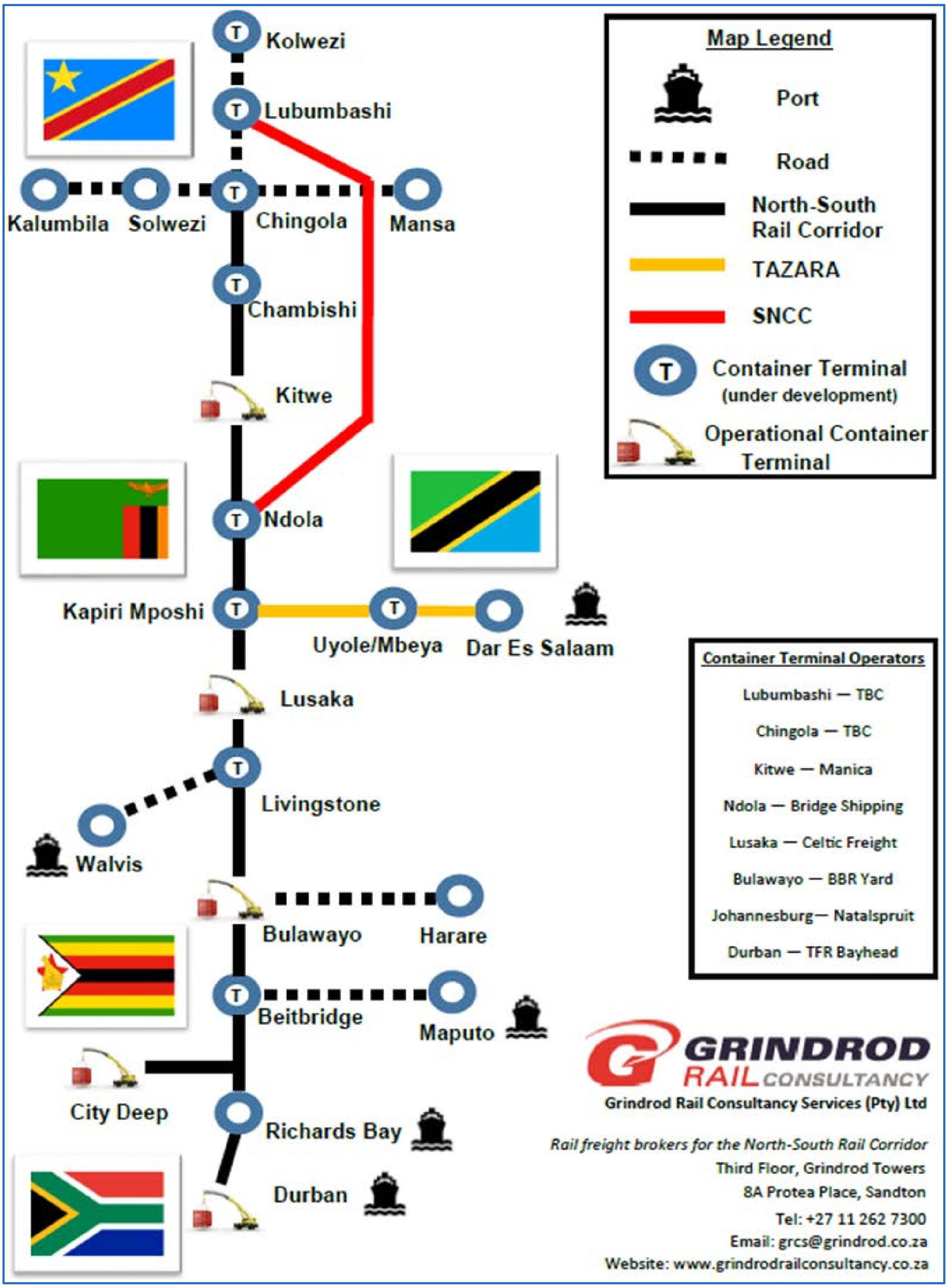


Figure by Grindrod, 2015.

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 x 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 x 30 m. This airport is largely serviced by Air Fast, providing four 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 fuelling infrastructure is available along all of the required routes to Kolwezi, albeit fuel quality and standards between countries are likely to vary. 50 ppm Sulfur fuel is readily available in the region. On site, it is planned for two or three fuel depots and filling stations to be owned by Kamoa and operated on a consignment basis by fuel suppliers.

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 nearby, 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 overflow 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 two 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 three 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

The overall water balance, including the bulk (raw) water requirement, is discussed in detail in Golder's report titled Kakula Feasibility Study Surface Water Report. On average, there is an excess of water at the Kakula site. The only water shortfalls (excluding water for domestic use) which must be supplied by raw water, occur during the concentrator and backfill plant start-up prior to the TSF return of water.

18.1.12.2 Bulk Water Requirements

Bulk (raw) water is obtained from the Western Wellfield, which is a high yielding and good quality water source. Water from the wellfield is pumped to a centralised transfer system located near the wellfield, which will transfer the raw water overland to the VD#2 potable water storage, and the process water dam located next to the process plant.

The use of raw water is minimised as far as possible by reusing excess underground mine water where feasible.

Raw water is mainly used in the following applications:

- Feed to the various potable water storage tanks, via chlorination units.
- Supplied to the backfill circuit in the start-up period in the absence of process water and gland service water produced by the concentrator.
- Make-up to the process water circuit via the process water dam.
- Wash bays.

- Make-up to the main decline mine service water tank (provisional only).

The capacity of the Western Wellfield is sufficient to supply the raw water requirements expected over life of mine.

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.

18.1.12.3 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

Three types of buildings will be used at Kakula:

- Modular buildings – These buildings are erected on a concrete slab. A galvanised channel base frame is fixed to the concrete, under external and internal walls. The prefabricated wall panels are screwed or riveted to the base channel and a similar wall frame connects the top of the panels. The doors, windows and roofing are installed in the same manner to the relevant panels. All wall switches, plug boxes, wiring, and fully equipped distribution boards are included.
- Steel structures with civil bases – sheeted steel with civil bases, plinths and surface beds. Typically, these buildings have filled-in brick work on the sides, brick offices and small stores on the inside to accommodate personnel. Roller shutter doors, standard doors and windows are included, together with power, lighting, general tools/equipment, furniture and an overhead crane (if needed). This building type is normally used for workshops and stores.
- Electrical Buildings – All MV substation buildings are made of structural steel with inverted box rib (IBR), roof sheeting, elevated concrete slab, and filled-in brick work. All LV substation buildings are shipped, containerised and supplied to site. Only civil bases will be cast on site with steel platforms for access.

18.1.13.1 Kakula Village

A permanent village has been constructed within the mine perimeter fence 1 km from the mine site to house expatriate and rotational DRC labour. The village have facilities and associated infrastructure to house 1,600 people.

Table 18.6 Modular Buildings

Description	Floor Area (m²)	Phase 1	Phase 2
Mine Lamp Room	3,072	✓	×
Change House Complex	4,308	×	✓
Gate House	78	✓	✓
Ablution #1	52	×	✓
VD #2 Offices	384	✓	✓
VD #2 Lamp Room	1,536	✓	×
VD #2 Security Building	52	✓	×
Vehicle Wash Bay LDV	20	×	✓
Tyre Change Area	120	×	✓
Concentrator #1 Control Room	975	✓	×
Plant Office	384	×	✓
Plant Entrance Gatehouse	65	✓	✓
Ablutions Building #1	128	×	✓
Plant Change House	192	×	✓
Offices - Main Office	3,127	✓	✓
Training and Induction Centre Terrace	1,408	✓	✓
EMS Complex	397	×	✓
Gate House - Main Entrance	480	✓	✓
Gate House - Truck Stop	52	✓	✓
Gate House - Bonded Yard	97	✓	×
Ablutions - Truck Stop	384	✓	✓
Ablutions - Bonded Yard	52	×	✓
Truck Stop Shops	192	×	✓
Laboratory Complex	2,500	✓	✓
Core Yard	570	×	✓
Offices - Main Weighbridge Office	52	✓	×

Table 18.7 Steel Structures with Civil Basis and Sheeting

Description	Floor Area (m²)	Phase 1	Phase 2
Engineering Workshop	1,919	×	✓
Heavy Vehicle Workshop	4,012	✓	✓
Tyre Store	570	×	✓
Mine Store	4,560	✓	✓
Sub-Component Store	570	×	✓
VD #2 Stores	1,140	✓	✓
VD #2 Workshop	1,140	✓	×
Plant Stores	253	✓	✓
Plant Workshop	1,545	✓	✓
Main Stores – Closed Store	4,560	×	✓
Surface Fleet Heavy Vehicle Workshop	3,210	✓	✓
Engineering Workshop	1,919	×	✓

18.1.14 Fire Protection System

A mine-wide fire reticulation system that adheres to Automatic Sprinkler Inspection Bureau (ASIB) 12th edition rules, was designed by an external consultant. The system will comprise pumped fire water ring mains on surface, a single gravity supply underground, deluge zones for all conveyors underground and on surface, hose reels, substation gas protection, and potable fire extinguishers.

18.1.15 Construction Facilities

Temporary construction facilities have been kept to a minimum by re-using offices, stores, and other facilities required for construction during the operational state. Construction facilities include the following items:

- Laydown area and construction stores.
- Construction power supply from SNEL and reticulation infrastructure.
- Offices for the mine and concentrator that will be used during operations.
- Stores.
- Multiple concrete batching plants - Contractor supplied.
- Soils/concrete laboratory.
- Construction cranage and light vehicles.

* Ablution facilities.
* Contractor’s camps.

Construction accommodation will be provided at the existing Kamoa Camp (350 beds) and a contractor’s camp will be constructed to house between 1,000–2,000 beds.

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 1,600 personnel in total of which only 1,000 will be constructed for the first 3.8 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 1,000–2,000 people at a time. The accommodation cost has to be covered in the contractors cost.

18.1.15.2 Construction Power

Construction power is to be supplied by a combination of local diesel generators provided by the contractors and construction distribution boards supplied from grid powered mini substations.

Provision has been made to reticulate grid power to the construction offices, the construction laydown areas and the concentrator plant construction areas from a series of mini substations. Remote areas, e.g. 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 Kansoko Site Infrastructure

18.2.1 Introduction

The Kansoko site infrastructure and Development Plan includes power supply, tailings dams, communications, logistics, transport options, materials, water and waste-water, buildings, accommodations, security, and medical services.

OreWin

18.2.2 Site Plan and Layout

A plan showing the locations of the mines and key infrastructure for Kakula and Kansoko is shown in Figure 18.12.

Figure 18.12 Site Conceptual Infrastructure Layout Plan

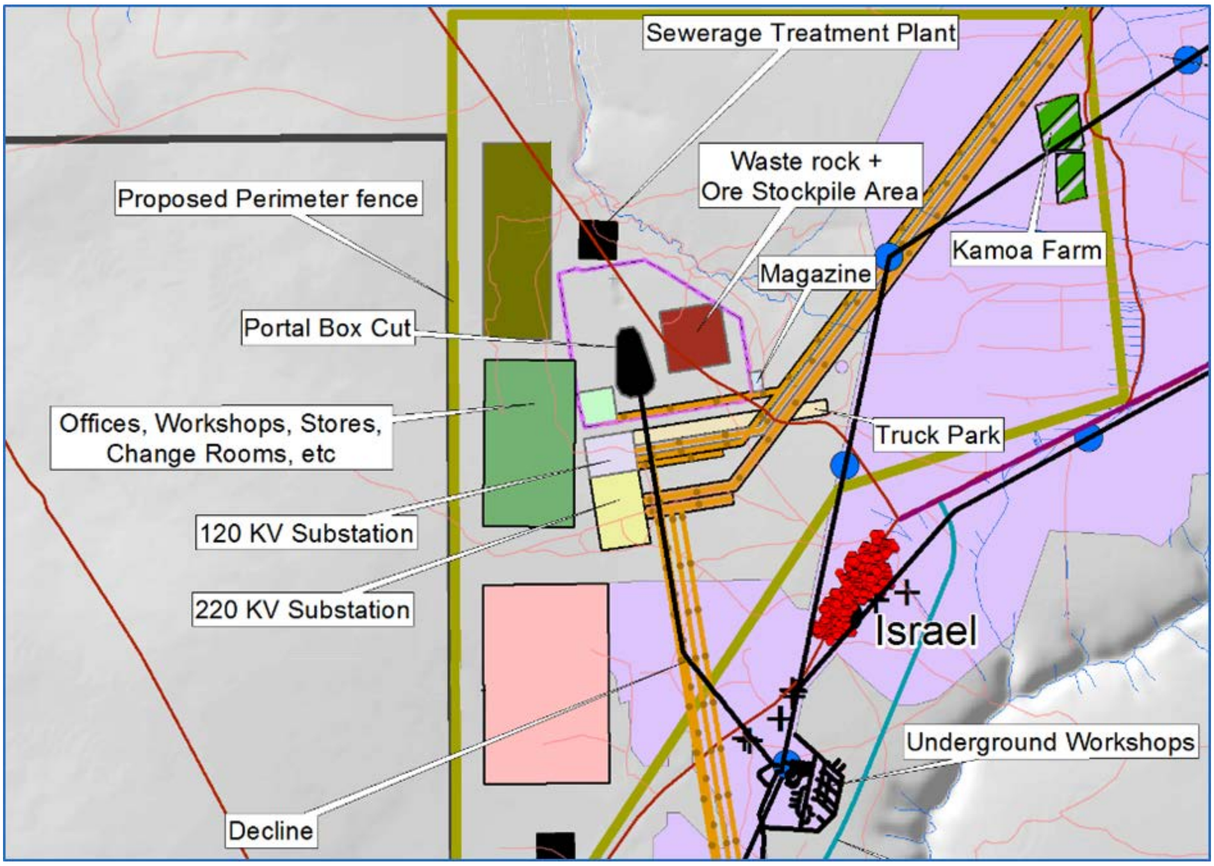


Figure by Ivanhoe, 2017.

The Kansoko site is compact and incorporates the crushing circuit, offices, electrical infrastructure, water infrastructure, surface mining offices and workshops, vehicle parking, warehouse 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 18.1.4.3 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. Optical ground wire (OPGW) 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 18.13 shows the design of new transmission lines and substations.

Figure 18.13 Planned Transmission Lines and Substations

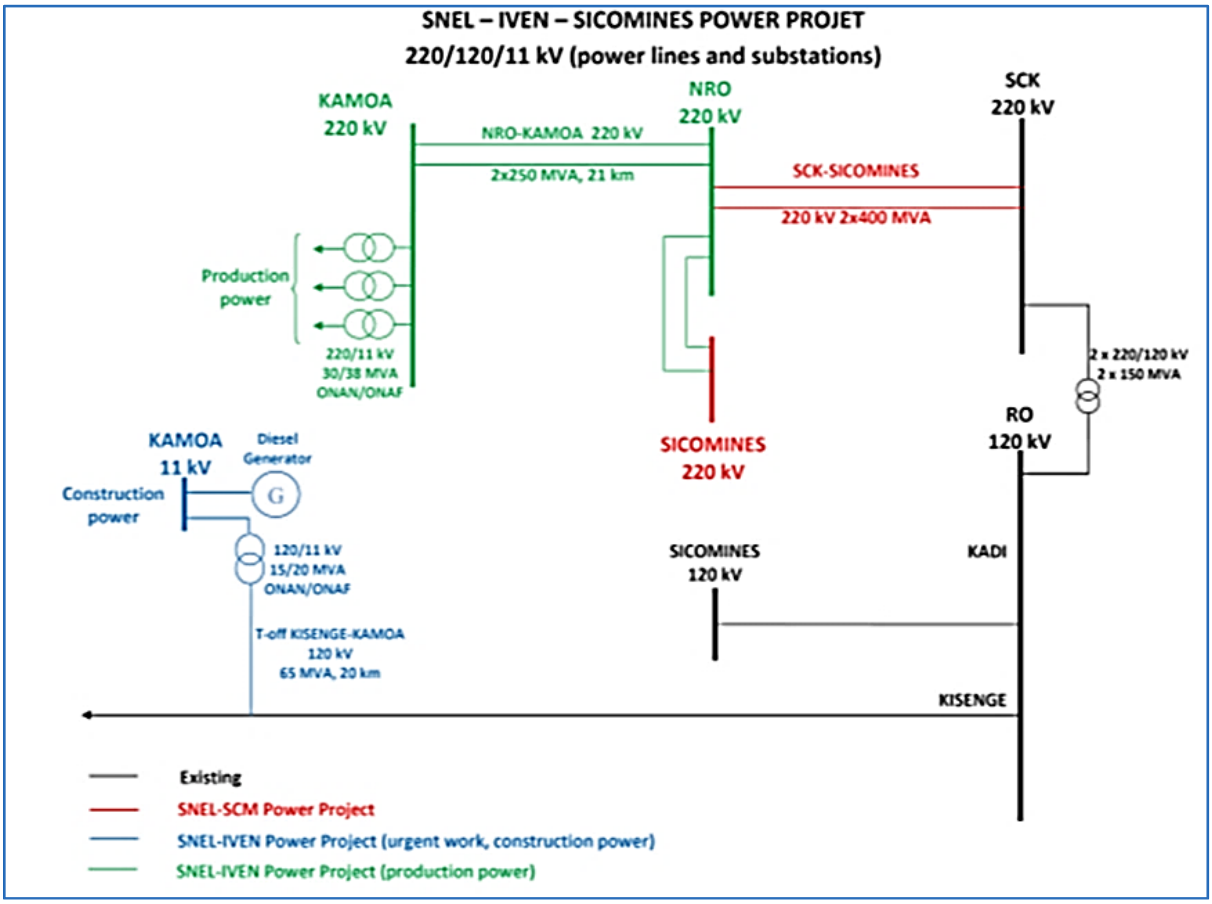


Figure by Ivanhoe, 2016.

18.2.4 Tailings Storage Facility

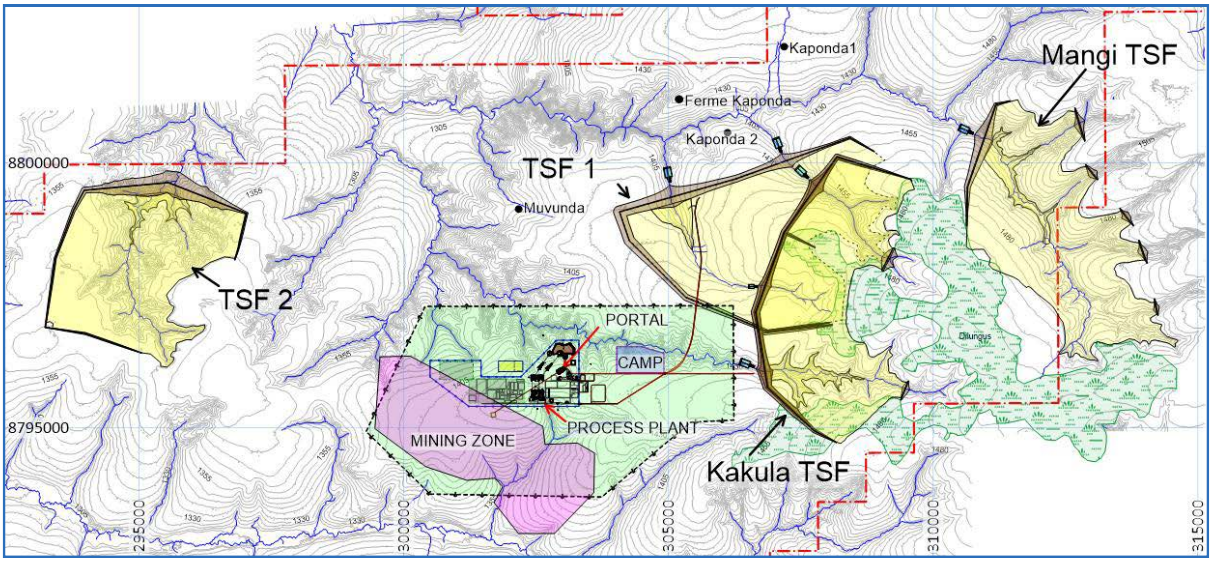
Epoch Resources (Pty) Ltd (Epoch) completed a basic design of the Tailings Storage Facility (TSF) and associated infrastructure as part of the Kakula-Kansoko 2020 PFS.

The terms of reference that Epoch was responsible for include:

- A Tailings Dam (TD) that accommodates 175 M dry tonnes of tailings over a 39-year LOM.
- The associated infrastructure for the TSF (i.e. perimeter slurry deposition pipeline, stormwater diversion trenches, pool access wall, toe paddocks 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.

A site selection Study was undertaken as part of the PEA in 2017. The required capacity for the study was for 228 Mt, to accommodate for possible further expansion. The preferred site was found to be the Kakula TSF site, shown in Figure 18.14.

Figure 18.14 TSF Site Selection



The layout of the TSF is shown in Figure 18.15 and the key design features of the facility are as follows:

- A Tailings Dam (TD);
  - Full containment, downstream construction method, with open-end deposition.
  - An engineered, earth-fill containment wall.
  - Curtain drain seepage collection system inside the containment wall (to reduce the phreatic level within the wall).
  - An HDPE liner over permeable soils.
- A pool access road along the centre of the valley.
- Emergency spillways at each phase of the Tailings Dam.
- Storm water diversion trenches upstream of the Tailings Dam.
- A floating pump decant system.
- An access road around the Tailings Dam.

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 18.15.

Figure 18.15 General Topography of the Preferred TSF Area

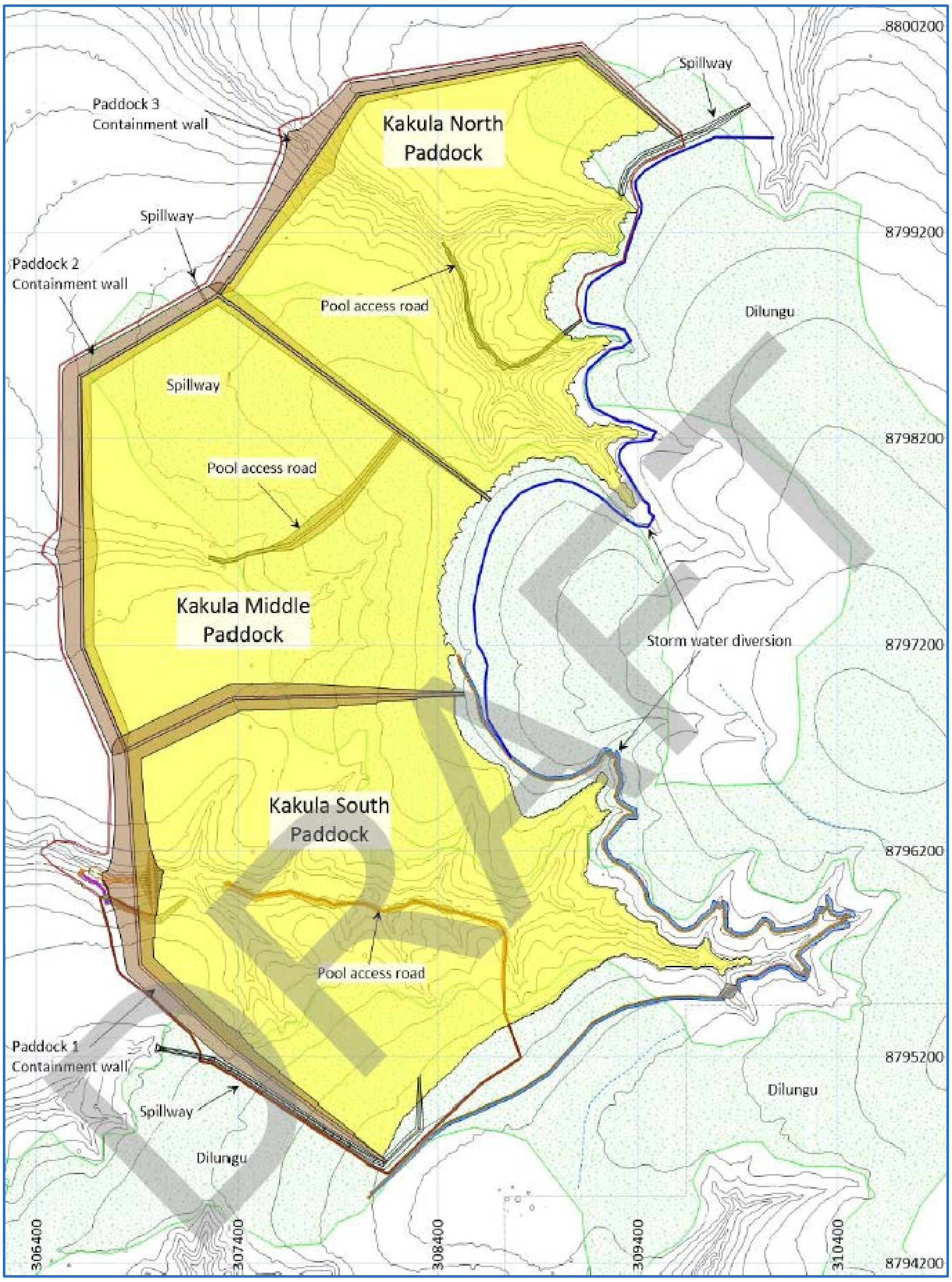


Figure by Epoch, 2020.

18.2.4.2 Design Criteria and Assumptions/Constraints

The design of the TSF was based on the design criteria shown in Table 18.8.

Table 18.8 Design Criteria Associated with Kansoko TSF

Description	Value	Unit
Design Life of Facility	39	years
Average Tailings Deposition Rate	4.4	Mtpa
Particle SG of Tailings product	2.92	
In-situ Void Ratio	1.0	
Particle size distribution of Tailings product	80% passing 66 micron	
Average dry density of tailings	1.28	t/m^3
Site's Seismicity	48	

18.2.4.3 Liner Requirements

Stability analyses were carried as part of the Feasibility Study of the Kakula South Paddock TSF. The results showed that the facility is stable under static and pseudo-static loads with factors of safety greater than 1.5 for static and 1.1 for pseudo-static loads, for all analyses that conform to the design. Extreme scenarios tested the effect of large pools, damaged drains and damaged liners, which aimed to show the efficacy of the drains and liner, as well as maintaining a small pool on the TD. It has been assumed that with a similar geometry, the middle and north paddocks will also have similar factors of safety.

18.2.4.4 TSF Site Selection

The Kakula TSF will comprise three compartments or paddocks over three valleys. Each compartment will be phased over a number of raises of the containment wall which will progress as a downstream facility in 5 m raises, therefore relying on the construction of an earth embankment or containment wall to store tailings. The southernmost paddock will be constructed first and comprises seven wall raises. The middle and northernmost paddocks will have four and three raises, respectively. The combined stage capacity curve for all three paddocks of the Kakula Tailings Dam, reflecting the relationship between tailings elevation, rate of rise, storage volume, footprint area, cumulative tonnage elevation and time was prepared (for both production cases). Figure 18.16 illustrates an example of the construction phasing, of a lined full containment facility, being implemented.

The first raise of the containment wall of the south paddock will be 20 m high (1,445 mamsl) which will provide a minimum of one year of capacity. Thereafter, the containment wall will be raised in 5 m lifts. The final wall height of the south paddock will be 50 m (1,475 mamsl) high. The final capacity of the TSF is approximately 6 Mt more than what was required for the PFS, however additional capacity has been provided to allow for possible future expansion of the mine. The final height of the wall may be revised during operations, reducing the wall height by 2.5 m, if required while still maintaining a 1.5 m freeboard. The containment wall and footprint has been phased in order to delay capital expenditure. The main items that can be phased are the liner, drainage systems and containment wall. The staged development of the Tailings Dam in shown in Figure 18.16. Yellow indicates tailings and brown indicates the containment wall.

Figure 18.16 Impoundment Wall and Self-Raise Lift Phasing

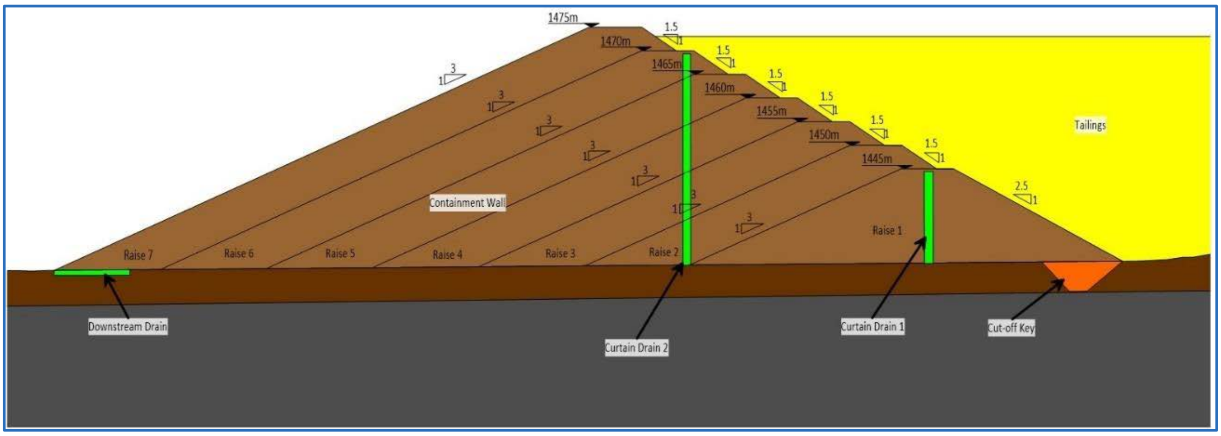


Figure by Epoch, 2020.

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.

18.2.4.5 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 1,465 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 1,500 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.
- 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.

18.2.4.6 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.

18.2.4.7 Water Balance

As part of the 6.0 Mtpa Feasibility Study of the south paddock, a water balance study was undertaken to assess the expected range of daily returns to the plant as well as the volume of excess water to be stored on the Tailings Dam. A water balance has not been undertaken for the PFS, however, as a significant portion of the return water volumes comprise rainwater, it may be assumed that the returns are similar for the two larger production rates, if the returns are presented as a percentage of the water sent to the TSF.

18.2.4.8 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.

18.2.4.9 Risks

The possible project risks associated with the current TSF design are as follows:

Tailings Storage Facilities pose a significant hazard to people and property around them as well as significant costs to the client. Specifically, they pose a risk to:

- Health and safety of workers, contractors and visitors to the mine.
- The environment (animals, plants, eco systems, habitats, wetlands etc.).
- The economic sustainability of the mining operation (business economics).
- The mine's reputation and relationship with the community (public, authorities, NGO's, neighbouring community).

The size and degree of the potential hazard depends on the location and size of the TD, site specific characteristics, method of construction, tailings material characteristics, construction materials, method of tailings dam development, operational control, closure planning and monitoring, and overall management.

18.2.4.10 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 Kansoko 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:

- 11 km Haul Road for the Kansoko crushing circuit to the Kakula Processing Facility. 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

The logistics for Kansoko are identical to those of Kakula outlined in Section 18.1.9.

18.2.9 Kansoko Overland Conveyor

A PFS design was done on a 6.0 Mtpa conveyor that will be installed to convey crushed rock from Kansoko mine to the Kakula concentrator complex once Kakula mining is completed. The conveyor is approximately 11 km long and is designed for daytime running only at a capacity of 1,420 t/hr. the conveyor route is illustrated in Figure 18.17.

Figure 18.17 Kansoko-Kakula Overland Conveyor

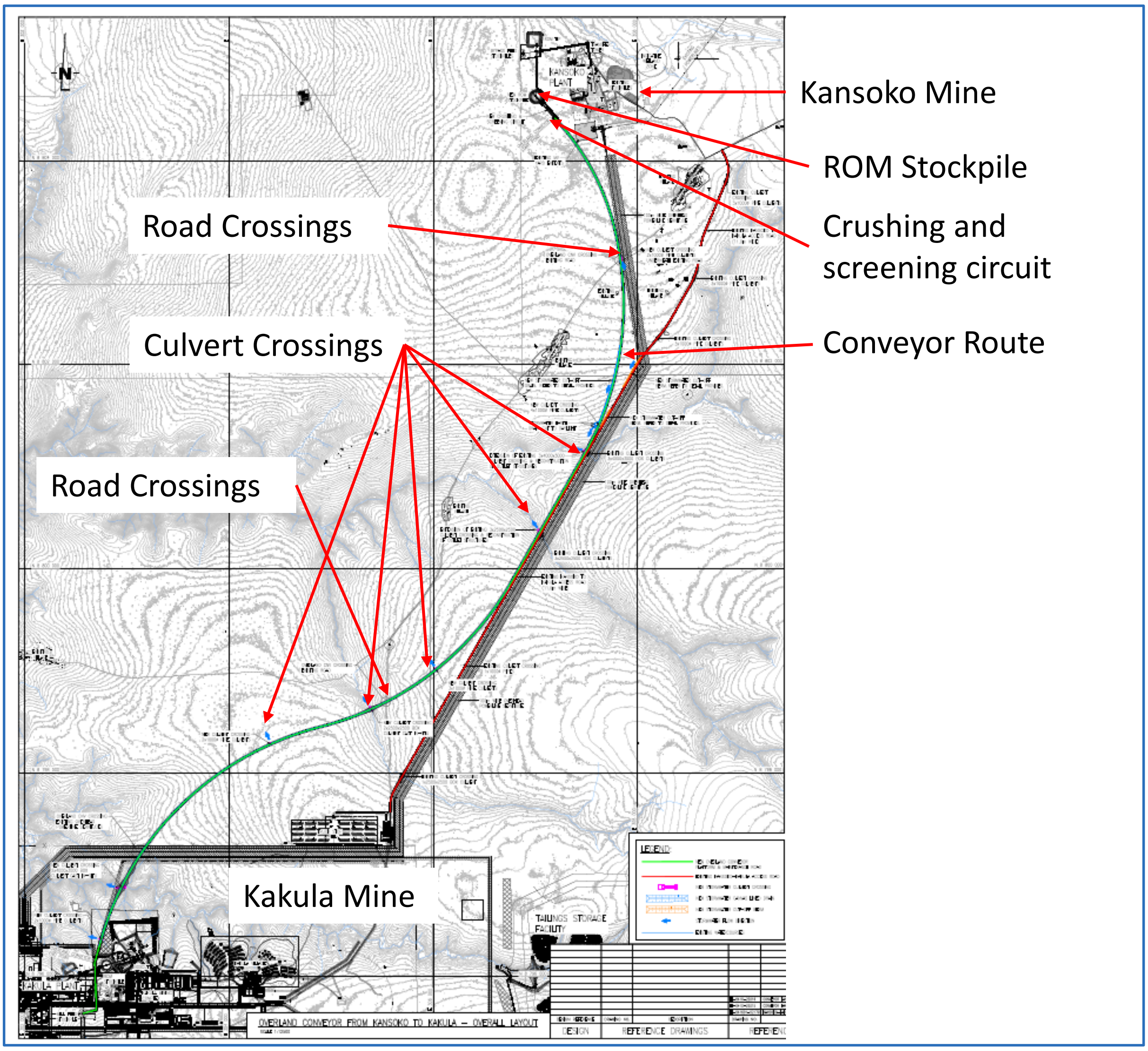


Figure By: DRA, 2020.

18.2.10 Airports

The Airport used by Kansoko is the same as Kakula outlined in Section 18.1.10.

18.2.11 Water and Wastewater Systems

18.2.11.1 Water Demand

The estimated water demand for the project scenario is given in Table 18.9. 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 18.9 Estimated Water Demand

Description	Units	Quantity
Mining Water Requirement	m^3/day	320
Potable Water Requirement	m^3/day	280
Contingency	%	10
Total Daily Requirement	m^3/day	660

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.

18.2.11.2 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 four boreholes (three production and one 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.

18.2.11.3 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.

18.2.11.4 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. and 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.

18.2.11.5 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 potential future plant, stockpile, and decline area is shown in Figure 18.18.

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 18.18 Stormwater Dam

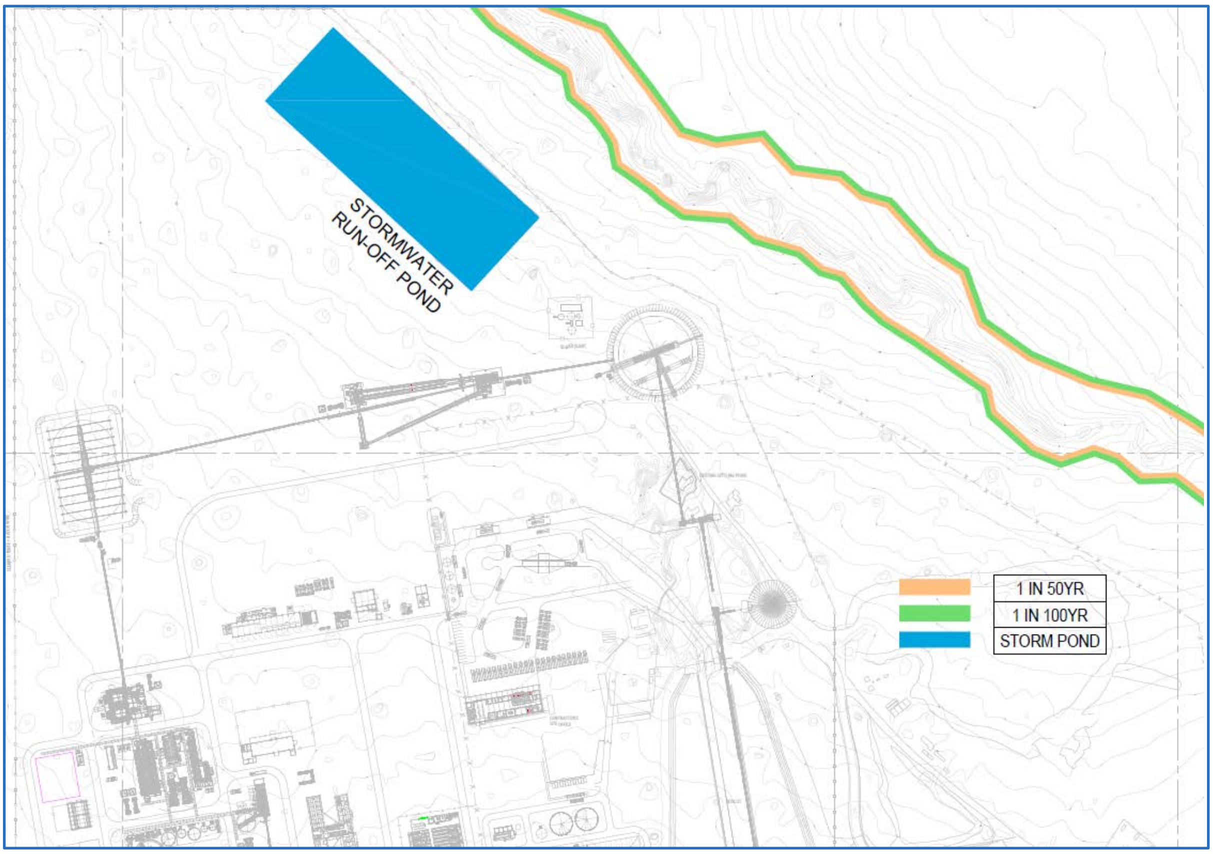


Figure by MDM, 2017.

18.2.11.6 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.

18.2.11.7 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.

18.2.12 Fire Protection and Detection

The fire protection and detection system for the surface 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. 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 specified as part of the overall Fire Protection System, 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.

18.2.13 Hospital and Medical Facilities

The clinic and first-aid facility will be that of the Kakula site.

18.2.14 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.

18.2.14.1 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).

18.2.15 Owner's Camp

18.2.15.1 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.

18.2.16 Construction Facilities

Temporary construction facilities have been kept to a minimum by re using offices, stores, and other facilities required for construction during the operational state. Construction facilities include the following items:

- Laydown area and construction stores.
- Construction power supply from SNEL and reticulation infrastructure.
- Offices for the mine and concentrator that will be used during operations.
- Stores.
- Multiple concrete batching plants.
- Soils/concrete laboratory.
- Construction cranage and light vehicles.
- Ablution facilities.
- Contractor's camps.

Construction accommodation will be provided at the existing Kamoa Camp (350 beds) and a contractor's camp will be constructed to house between 1,000–2,000 beds.

18.3 Comments on Section 18

18.3.1 Kakula Infrastructure

Infrastructure planning was completed at an appropriate level of accuracy for the Kakula 2020 FS 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 Kakula-Kansoko 2020 PFS and no issues were identified that will have a material negative impact upon the financial viability of the project.

19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies and Offtake Strategy

IDP20 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 market outlook for TC $62/dmt concentrate and RC $0.062/lb Cu. The payable copper for concentrate is estimated at 96.75%, based on discussions with potential off takers. The base case for IDP20 assumes a copper price of US$2.95/lb in 2021 and US$3.10/lb long-term, on a real basis, and is consistent with long term estimates and pricing used in other published studies.

For the purposes of the IDP20 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, China Nonferrous Metal Mining Group (CNMC) recently completed a smelter, Lualaba Copper Smelter. A two phase commissioning is planned with a concentrate capacity of 500 kt (dry) in each stage. This smelter will rely on concentrates from the DRC but the two 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. Lualaba Copper Smelter 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 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 summarised as emerging market demand, developed market infrastructure and housing demand, 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 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. Infrastructure spending in China has been the primary driving force behind the global demand for copper that is being fuelled by the trend of urbanisation in the country. Coupled with infrastructure spending is the increased spending to construct a greener economy in order to meet decarbonisation targets, which also favours copper as it is an essential metal for both electric vehicles and renewable energy technologies. A study conducted by the European Copper Institute shows that one tonne of copper can save 100–7,500 t of CO_2 emissions per year. 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 rest-of-world ROW adoption rates.

On the supply side, copper output from the world's largest copper miners in South America has been disrupted following multiple months of lockdown and labour disruptions from the impact of the COVID-19 pandemic, which is expected to decrease refined copper output by 2.1% in 2020. Long term copper supply remains constrained due to declining exploration spending and low discovery rates of major new copper projects. Existing mines continue to exhibit declining ore grades, with average copper grade mined across the world falling by more than 40% since 2000, resulting in significant additional investments required in expanding processing capacity. As a result of these trends, the global production pipeline lags that of global demand. Fitch Solutions expects the copper market deficit to increase from 299,000 t in 2020 to 489,000 t by 2024 and further to 510,000 t by 2027.

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 AND SOCIAL

20.1 Introduction

Ivanhoe Mines Ltd, Kamoa Copper SA (Kamoa) appointed Golder Associates Africa (Pty) Ltd (Golder) to compile the environmental and social inputs as part of the feasibility study for the Kakula Mine Licence area. Golder has previously contributed environmental and social inputs to the prefeasibility study for the Kakula Mine in 2018.

The environmental and social work is reported in:

- Golder, 2020. Kakula 6.0 Mtpa Feasibility Study, Environmental and Social Inputs. Golder Associates Africa (Pty) Ltd. Report No. 19129784 331400 4 issued 21 May 2020.

The above report includes as appendices, two additional reports:

- Golder, 2020. Kamoa-Kakula Project: Greenhouse Gas Emissions Assessment. Golder Associates Africa (Pty) Ltd. Report No. 19129784 331986 5 issued May 2020.
- Golder, 2020. Biodiversity Screening Study and Ecosystem Services Overview in line with The IFC Standards. Golder Associates Africa (Pty) Ltd. Report No. 19129784 330544 2 issued May 2020.

20.2 Previous Work

An initial environmental and social impact study (ESIS) authored by African Mining Consultants (2011) was submitted by Kamoa Copper SA (then known as African Minerals (Barbados) Limited) in support of a mining licence, which was granted in August 2012.

An update of the ESIS was commenced by Golder in 2012 with the collection of environmental, social and health data, stakeholder consultation and the development of a detailed scoping report. The work was put on hold in Q2’14, pending finalisation of the project design, and resumed in Q3’16 with the completion of an environmental and social management plan. The update was completed, submitted and approved as an environmental impact study (EIS) in Q1’17.

Several additional environmental impact studies have been undertaken since 2013 for supporting infrastructure such as power transmission lines.

A change to the project description since 2017 have resulted in an updated environmental and social impact assessment (ESIA) authored by CEMIC (2019) and approved on 14 July 2020.

20.3 Policies Regulatory and Administrative Framework

Through the Environment Policy, Kamoa Copper SA commits to comply with DRC environmental legislation and standards in environmental management for its exploration and operations activities. It aims to ensure the long term sustainability and minimal degradation and impact on surrounding ecosystems in all areas of operation.

Kamoa Copper SA are required to comply with a range of environmental and social international conventions and agreements which the DRC is a party to.

Kamoa Copper SA seeks to comply with the following international organisations:

- The Equator Principles.
- The International Finance Corporation (IFC) Performance Standards.
- The World Bank Group Environmental Health and Safety (EHS) Guidelines.
- International Association for Public Participation (IAP2).

A review of the environmental and social legislation relevant to Kamoa identified the following national authorities with jurisdiction over reviewing the EIS and granting a renewed mining licence:

- The Minister of Mines.
- The Department for the Protection of the Mining Environment (DPEM).
- The Directorate of Mines.
- The Mining Registry (Cadastre Minier – CAMI).
- The ACE (Congolese Agency for Environment).

20.4 Stakeholder Engagement

20.4.1 Public Participation

Stakeholder engagement has occurred with affected people, government and traditional authorities, and other interested and concerned people since the start of exploration is 2004. This engagement took place by way of meetings with community leaders, full community meetings, and focus group meetings. Engagement was by company representatives, and in particular community engagement and environmental personnel. The focus of the engagement was:

- Overview of company policies and intent.
- Notification of drilling and other activities on the site, and of the process of compensation for damages.
- Health and safety instruction pertaining to site activities.
- Community development support by the company.

Public consultation was undertaken during the Exploitation Permit application process in 2010–2011.

The pre scoping phase in 2012 of the ESIA update included capacity building to ensure a “free, prior and informed” process. More than 450 issues were raised during the capacity building phase. These issues were categorised into; agricultural and other support, employment, resettlement, education, health, transparency, trust and credibility, noise, recruitment policy and principles.

Further pubic consultations were carried out as part of ESIA updates in April 2013, October 2016, and May 2019.

20.4.2 Ongoing Stakeholder Engagement

Through the development and implementation of a stakeholder engagement plan (SEP), Kamoa Copper SA aims at establishing, maintaining and preserving a transparent and beneficial ongoing relationship with various stakeholders during the lifetime of the project. The stakeholders include Kamoa management, the Sustainability department (which have been identified including Government officials), impacted communities, traditional chiefs, CBOs, project personnel, civil society, trade unions, trained community liaison teams, community development committees, local development committee, and a stakeholder database and matrix. A stakeholder matrix designed to map the level of influence of key stakeholders on Kamoa and the level of public consultation that should be targeted, is in place. This database has been updated annually, and the team keeps meticulous record of all engagements and issues raised during engagements for purposes of follow up and management.

The SEP was designed to meet international standards as determined in the Equator Principles (Equator Principles, 2012), IFC Performance Standards (IFC, 2012) and best practice guidelines for stakeholder engagement as prescribed by the International Association for Public Participation (IAP2).

A grievance mechanism to handle external complaints and social incidents was developed and put in place in 2019. Most grievances to date relate to roads damaged by the increase in traffic and exploration activities, damages to crops and fields, and request for compensation due to road incidents (goats, chickens, dogs).

20.4.3 Sustainable Development Plan

The last socio-economic survey performed by Golder in 2014 indicated that the Kamoa community is characterised by high levels of unemployment and poverty. Unsustainable forest clearance for charcoal production is destroying the fragile eco system that provides emergency food security to the local community. Uncontrolled fire and unsustainable trapping and hunting for the urban bush meat market are further depleting natural resources.

There is a need for a sustainable development program that will improve food security and the standard of living of the local communities whilst conserving the natural environment throughout the mine's lifecycle and beyond.

Kamoa Copper SA compiled a high level sustainable development plan (SDP) as part of its ESIA, aiming at improving the economic, social and cultural well-being of the local populations affected by the project during and after the mine operation in partnership with specialised government services and non-profit organisations. Following the current reviewed mining code, Kamoa has developed with the impacted communities a Social Plan "cahier des charges" which describes a list of five years of social projects that will be implemented in the Kamoa footprint. The estimated budget for the cahier des charges is US$8.6 M over the five years to start from the effective commercial production.

Kamoa has launched several projects, in the following sectors:

- Social projects for local communities: education and adult literacy, health and a sewing training programme.
- Community infrastructure: schools, community water wells, donations to various health centres, construction of houses as part of relocation, and construction of or financing access roads.
- Community development projects: assistance with initiatives identified in Lufupa and Luilu local development programs, entrepreneurship projects, and Kamoa Sustainable Livelihoods Project.

The Kamoa Sustainable Livelihoods Project supports initiatives such as: small scale maize production, market gardening, poultry production, bee farming, aquaculture, and fruit trees.

20.5 Environmental and Social Management System

An Environmental and Social Management System has been developed and partially implemented at Kamoa in order to manage significant risks, such as aspects with a high impact on the environment. Kamoa is currently implementing a custom designed software (Isometrix software) that will assist in the management of environment and social performance data and will facilitate monitoring and reporting for the Kamoa project.

20.6 Key Environmental and Social Sensitivities

20.6.1 Radiation

A radiological assessment was undertaken by the Nuclear Energy Corporation of South Africa (NECSA) in March 2013. The results indicated normal radiation conditions and limited radiological risk.

A second assessment was carried out in June 2019 as part of the ESIA update undertaken by CEMIC. The results are comparable to the radiation levels measured during the 2013 radiological assessment. Current mining activities pose no enhanced background radiological risk to the public.

20.6.2 Land Utilisation

A detailed soils, land use and land capability study was undertaken by Golder in 2012–2013. A second survey was carried out by CEMIC as part of the 2019 ESIA update for the Kamoa mining operations.

Land use generally correlates to soil types and topography but is also influenced by access to water, access to roads and proximity to major settlements such as Kolwezi. Most of the soils in hilly areas and Dilungus remain under natural vegetation. The main traditional land uses include subsistence cropping (predominantly cassava, which is a less demanding crop for the poor and depleted soils) and logging of wood for charcoal production. The Kamoa-Kakula Project will utilise approximately 30% of the mining lease area for mining activities with ~1% used for farming projects. Approximately 50% of the licence area was identified to be in a natural state, with the remaining 20% representing degraded/deforested miombo (modified habitat).

The natural fertility of soils in the project area is marginal with most areas having poor to limited agriculture potential.

20.6.3 Air and Water Monitoring

Monitoring programmes are ongoing for stream water flow and water quality. A baseline programme commenced in 2010, ceased in 2015, and was re-established in 2018. The data obtained during the 2018–2019 monitoring period indicates a good surface water quality as all the parameters are generally well within the target limits, apart from electrical conductivity measurements ranging from 1–30 mS/m. Total dissolved solids have a pronounced trend in most samples, presumably associated with the dry season.

As per Annex VIII of the DRC mining regulations (2018), the effluent quality and flow discharges is required to be monitored. The mine started monitoring the effluent quality at Kakula North (main decline) discharge in July 2019 and at Kakula South discharge in November 2019. In July 2019, the monitoring of the Kakula main decline and ventilation south decline mine waters indicated that TSS remained the major concern for Kakula mine water to be discharged into the natural environment. Monitoring of groundwater levels commenced in 2018. The groundwater quality in the Kakula Area is a Ca/Mg–HCO_3 water type and classified as fresh water with a high degree of seasonal replenishment. This water quality type is similar to rainwater and generally well within the DRC and World Health Organisation (WHO) drinking water limits.

Air quality monitoring records for the Kamoa-Kakula project extend back to approximately 2013.

A number of dust fallout or air pollutant surveys have been undertaken by Golder in 2018 and 2019, Compass Green Worldwide in 2018, and CEMIC in 2019.

Most air pollutants fall within or well below the standards except for dust fallout. Fine dust (PM10 and PM2.5) particulates show exceedances relatively frequently and predominantly during the dry season.

20.6.4 Biological Environment

20.6.4.1 Flora Surveys

The characterisation of the terrestrial ecology is based on five field surveys – three surveys undertaken by Golder between November 2011 and July 2013, a survey of the Kakula area by Golder in 2016 and a survey carried out in June 2019 by CEMIC.

Vegetation communities are categorised under three broad structural formations: forest, woodland, and grassland and include:

- Miombo Woodland: natural, hillslope; degraded, or fragmented.
- Uapaca Fringe Woodland.
- Riparian Forest.
- Hygrophilous Grassland.

Large scale clearing of woodland by local communities for agriculture and charcoal production has occurred, and continues to occur. Fire is also widely applied and has in some areas maintained vegetation in an open, secondary form. The woodland is interspersed with Dambos (valley bottom grassy wetlands), which may constitute up to 30% of the region.

The tall Semi-deciduous Woodland community, Riparian Forest and Watershed Plains are species rich. A total of 230 plant species were recorded during the various field surveys, of these species only three are currently listed as IUCN listed Red Data flora species. Eight exotic species were also recorded in the study area. No Protected Plant Species listed in Article 6 of Annex XII of the DRC Mining regulations were found in the study area. Sensitive areas include shrublands, Dambos, Dilungus (unique ecosystems which have ecological links with Dambos) and the Miombo forest to the east of the Project area.

20.6.5 Fauna Surveys

Mammal species diversity is considered moderate in the study area and it is highly unlikely to find many of the larger mammalian taxa considering the degree of habitat modification and subsistence hunting throughout the study area. Twenty three reptilian species were also recorded during the various surveys – these species are not restricted in terms of habitat or distribution and none of the species recorded are classified as Red Data species or protected as per DRC legislation.

A total of 156 bird species were found to occur within the study area during the time of the study, mostly recorded in the Tall Semi deciduous Woodland and in the Secondary Woodland. Three bird species of conservation importance were recorded during the 2016 field visit, namely Wattled Crane and Secretary Bird (vulnerable), and Bateleur Eagle (near threatened).

The results of surveys of aquatic fauna (fish, amphibians, macro invertebrates) undertaken in the study area indicated a healthy population, and a diverse community with 33 species observed. Based on the current information no red listed fish species occur within the area. Eleven amphibian and twenty-three macro-invertebrate species were also recorded during the various surveys. Instream habitats integrity was generally found to be natural with some localised disturbances from local villages utilising the resource.

20.6.5.1 Critical Habitat and Ecosystem Services

Golder undertook a critical habitat (CH) screening study and an ecosystem services overview of the Kakula project area. Based on a defined area of analysis, the biodiversity features that may constrain the future mining activities have been discussed in relation to:

- Biodiversity features that could potentially trigger critical habitat constraints.
- Natural and modified habitats that support high biodiversity values.
- Ecosystems that may support potentially irreplaceable or vulnerable species, habitats and ecosystem services.

A number of species with potential to trigger CH under criteria 1–3 were identified in this screening study. Peatlands and Pyrophytic Watershed Grasslands (Dilungu's) were screened as potential CH (Criteria 4). A total of 113.5 ha and 4,263 ha of peatlands and Pyrophytic Watershed Grasslands respectively were identified within the Kamoa Mine Concession Area, of which 2.9 ha (2.5%) and 140.7 ha (3.3.%) of each unique ecosystem is located within the proposed infrastructure/tailings dam north and south respectively.

From an ecosystem services perspective, several provision, regulating, cultural and supporting ecosystem services were identified within the woodland forest, grasslands, riparian areas and rivers, depended on by both local communities within the Concession Area, as well as regional downstream users.

20.6.5.2 Water Management

The water demand for Kamoa Copper is estimated to be 9.1 ML/d for the 6.0 Mtpa mine. For the Kakula mine, a wellfield has been identified 3 km south-west of the mine to supply the mine with sufficient, good quality borehole water.

Further information on water management is contained in Section 18.

20.6.5.3 Waste Management

Information on the tailings waste characterisation and tailing storage facility is contained in Section 18.

The mine will generate large amounts of general waste as well as industrial wastes. These will be sorted to facilitate reuse and recycling. All industrial or hazardous waste will be stored in secure areas. Medical and clinic waste will be incinerated. A landfill site facility will be used for non-hazardous wastes generated at the mine.

20.6.6 Social Environment

The description of the social environment is based on several surveys undertaken in the project area between 2010–2019.

The project area is characterised by scattered, undeveloped rural villages and hamlets divided between the two groupings of Mwilu and Musokantanda. The most common ethnic group is the Ndembo, which accounts for 37% of the population, followed by Sanga, which represent 35%.

A total of 40 villages or communities fall within the mine concession area, with a population estimated at 10,348 people, and 22 of those villages would potentially be affected by the mining operations. The Kakula study area falls within the Musokantanda grouping.

Agriculture (maize and cassava) is the main economic activity. Livelihoods are primarily dependent on subsistence farming and charcoal burning, supplemented by limited formal and informal employment. Natural resources (water, wood, flora, and fauna) play a vital role in the lives of residents in the project area, in water supply, energy provision, income generation, nutritional and health benefits.

Demand for charcoal is high, both amongst villages in the project area, and more importantly in Kolwezi due to a rapid population growth and limited electricity provision. The consequence of a high demand for charcoal is a high rate of deforestation.

Formal employment opportunities are very low in the local study area. The presence of mining companies and service providers at various mining sites is an important source of income in the area; however, only small numbers of people have had formal employment in the past 12 months and only limited number of locals have been employed by Kamoa Copper SA, or by contractors of Kamoa Copper SA.

Artisanal mining also remains a subsistence activity in the Kolwezi region and continues to attract many miners to the region. This informal sector usually involves individuals or cooperatives that exploit minerals such as copper and cobalt. As concessions are granted to large mining companies, illegal artisanal miners are under increasing pressure, which results in (often violent) confrontations with law enforcement and mining company employees. Large scale mining operators are faced with a challenging environment in manage impacts and risks related to illegal artisanal and small scale mining within existing exploitation permits. However, there is no presence of artisanal miners in the immediate neighbouring of Kamoa project footprint.

The Kakula area generally has average levels of income, with higher levels of income derived from charcoal production. The project is likely to have a negligible impact on this village group – provided that the sources of income are maintained. The Kamoa livelihoods and local economic programs have shown significant household incomes for lead farmers and local entrepreneurs involved in vegetable, maize, fishing and brick production. There is a huge need to expand these initiatives to more community members given the high demand which are being recorded.

20.6.6.1 Indigenous Peoples

Information sourced from secondary data review and qualitative research undertaken in 2012 indicated a low possibility of the presence of Indigenous Peoples. Given the extent of population movement and migratory patterns, the presence of communities with distinct identity, customs or language in the local study area is therefore highly unlikely.

20.6.6.2 Community Health and Safety

Specific health information was collected for the Project in 2013 by Golder and in 2019 by CEMIC. Results of the surveys indicated that health infrastructures are insufficient to meet the demand of the populations in the area and that the most common disease in the project area is malaria.

A health impact scoping assessment was undertaken by Golder in 2012 the potential risks to community health, safety and security related to: potential contamination of drinking water (low risk), transport and deliveries of large quantities of hazardous materials to the mine (low risk), potential impacts on water availability due to the construction of the tailings storage facility on two Dilungu areas (risk to be assessed), exposure to diseases (risk to be assessed), and risks of altercations between security providers and communities.

Noise Monitoring

DRC national legislation for permitted noise levels due to mining is set out in the DRC Mining regulations 2018. Noise monitoring records for the Kamoa-Kakula project extend back to approximately 2010 and include:

- Baseline noise monitoring by African Mining Consultants on seven occasions between November 2010 and September 2012.
- Baseline noise monitoring by Golder at 18 sites during 2014.
- Baseline noise monitoring by CEMIC in 2019.
- Noise monitoring on a monthly basis by Kamoa Copper SA since March 2019.

The results show that the daytime ambient levels in local communities are significantly elevated and typically exceed the DRC daytime limit most of the time. Noise levels measured at the mine infrastructure are generally below the limit for industrial areas (70 dB(A)), except for noise levels at the Kakula North box-cut, North emergency ventilation and workshop area, which exceed the limit most of the time. It must however be noted that the noise level results may have been misrepresented due to an inadequate noise monitoring methodology.

In order to meet international standards and guidelines, it is recommended that:

- A consolidated noise management plan be developed for the wider mining rights area.
- The noise monitoring campaign be improved to allow compliance with the DRC requirements and recommendations of the specialists under the ESIA.

Vibration monitoring

DRC national legislation for permitted vibration levels is set out in the DRC Mining regulations 2018. Criteria for ground and airborne vibrations are proposed for the mining operation.

A pre-blasting survey within a 1,000 m blasting zone was undertaken in 2013 to assess the potential impact zones of blasting to the surrounding communities. As the blasting is underground, there is little to no risk of airborne vibrations impacting on nearby sensitive receptors however ground vibration may have an impact. CEMIC undertook ground vibration monitoring within the communities and at some of the operational infrastructure in 2019. The average ground vibration level observed within the communities was 6.5 mm/s (DRC vibration limit of 12.5 mm/s). Although this level falls within the DRC limit, considering that most of the residences in the nearby villages are of traditional construction type, they are likely to be more sensitive to blast vibrations than structures using modern materials and technology.

20.6.6.3 Physical and Economic Displacement

Kamoa has developed a draft resettlement and rehabilitation policy framework guided by national legislation and IFC Performance Standards and International Council of Mining and Metals guidelines. The aim of the policy framework is to avoid, and when avoidance is not possible, minimise displacement by exploring alternative project designs and minimising adverse social and economic impacts from land acquisition or restrictions on land use.

The first phase of the resettlement process (Phase 1) targeted population affected by the construction of the Kakula mine fence (Exclusion Zone). Alternative Plus, a Congolese NGO facilitated the process, including the gathering of the relevant baseline data, delineation and type of each structure or field that would be affected, and stakeholder engagement. Phase 1 affected 45 households in 15 hamlets, relocated to Muvunda village in 2018, 115 fields, and 678 fruit trees.

The Phase 2 Resettlement Action Plan was compiled by the NGO Alternatives Plus in July 2019 and concerned local communities affected by the construction of vent shafts. Phase 2 consists of 64 households in seven hamlets, 955 fields, and 13,098 fruit trees.

Resettlement was also required for the Kamoa to Kolweze Airport Road which is a provincial responsibility. Resettlement for the Kansoko to Kakula Bypass Israel Road requires 12 households, 10 fields, and 31 fruit trees to be delocalised.

The Phase 3A resettlement plan was compiled by NGO Alternative Plus in November 2019 and concerned local communities that will be affected by the construction of the tailings storage facility and sand quarry. Phase 3A affected 19 households in four hamlets, 117 fields and 1,071 fruit trees.

The Phase 3B resettlement plan was compiled by NGO Altnerative Plus in February 2020 and concerned local communities that will be affected by the construction of the tailings storage facility. Phase 3B affected 5 households in one hamlet, 125 fields and 1,045 fruit trees.

20.6.6.4 Cultural and Archaeological

Archaeological and cultural heritage surveys were conducted in the project area in 2010, 2012, 2013, 2016 and 2019 as part of the original EIA and subsequent updates.

The results of these surveys showed that the archaeological potential is poor or of low importance throughout the study area. Cultural heritage features in the study area include cemeteries, sacred and ceremonial places, historical context and beliefs, an old smelting furnace and traditional practices.

The overwhelming majority of cemeteries are located between Musokantanda and Israel, along the road linking the main villages. The following sacred, cultural and ceremonial places where identified by local communities in the Kakula study area:

- Rivers and waterfalls – the largest waterfall (Lufuba) is considered a sacred place, the other falls are used as ceremonial places.
- Muyombo (sacred trees) are located at Musokantanda and Muvunda villages.
- Springs classified as sacred at Musokantanda.

The conflict potential based on the historical dynamic between Chief Mwilu and Musokantanda was found to be unlikely. Cultural practices have been affected in contemporary times by various movements of people in and out prior to mining activities, and modernisation. However, the ancestral belief system is an important aspect of local culture and several key aspects relating to local custom and culture must be noted and considered. To date, graves and sacred sites have been avoided.

The following recommendations are made:

- Compile and implement a heritage management plan.
- In the event of a grave needing to be relocated, the relocation process will be included in the resettlement plan.
- A Phase 2 archaeological survey of the plant area due to the limitations of the baseline studies and the location of a potential archaeological site directly north of the proposed plant area.

20.7 Analysis of Key Impacts

Since 2012, after the approval of the original EIS (AMC, 2012), Kamoa has continued with exploration work, completed initial construction of box cuts, declines, dewatering and ventilation infrastructure, settling dams, and support infrastructure at both Kansoko and Kakula.

Earthworks at Kakula started in March 2019. Currently the Kakula development has caused several impacts associated with land clearance, some disruption to surface water runoff and flow; discharge of mine water to the environment; construction noise; air emissions; and physical and economic displacements of people.

Drilling and continued exploration has resulted in:

- Land clearance for road access and drill pads.
- Some small incidents of soil contamination due to spills.
- Dust from exposed areas, excavation, drilling and traffic as well as vehicle tail pipe emissions.
- Soil erosion due to clearance of land and establishment of roads, particularly in the Dilungus.
- Some traffic and construction noise and vibration impacts on sensitive receptors.
- No significant trace of impacts on surface and groundwater have been identified through ongoing environmental monitoring.

20.7.1 Physical Environment

The key physical impacts associated with the construction and operation of the Kakula Mine include:

- Topographic changes such as for waste rock stockpiles / TSF, and potential subsidence.
- Increased dust and noise levels.
- Impacts on surface water due to site clearance, surface infrastructure, effluent / mine water discharge or potential contamination from the tailings facility in event of a failure of the TSF.
- Potential groundwater contamination, and reduction in groundwater availability due to dewatering.

20.7.2 Biological Environment

The main impact on the biological environment due to the mine is the loss of potential critical habitat. Additional studies are required to determine the extent of critical habitats of peatlands and watershed grasslands (Dilungus). Furthermore, given the important role of the Dilungus as sources of baseflow for streams, impacts to surface water and groundwater related to the disruption of the Dilungus need to be assessed further.

Additional impacts on the biological environment by the Kakula Mine include:

- Increase in exotic or invader species.
- Clearing for site access and infrastructure causing the direct loss and fragmentation of riparian and watershed habitats.
- Degradation from dust reducing air quality.
- Degradation from increased human access increasing pressure on terrestrial habitats.

20.7.3 Socio-Economic Environment

The key impacts on the social environment, associated with the Kakula Mine include:

- Physical and economic displacement of households.
- The project will lead to population growth, through an in migration of project employees and opportunity seekers.
- Increased pressure on already limited basic services and infrastructure.
- Relocation of cemeteries due to project development.

20.7.4 Greenhouse Gas Intensity Metric and Benchmarking

- In September 2020, Kamoa commissioned Hatch Africa (Pty) Ltd (Hatch) to compile the Kamoa data received from Ivanhoe and other third party contributors and public available data in the form of a report as a desktop study for Greenhouse gas (GHG) inventory and intensity metric estimate calculation for copper to be produced at the Phase 1 Kakula underground copper mine and surface processing complex at its Kamoa-Kakula Project.
- Emission sources considered included heavy machinery and passenger vehicles used in the mine and around the concentrator; an emergency diesel generator; an incinerator; mining explosives; and purchased electricity from the national electricity provider, Société Nationale d'Élecetricité (SNEL), and formed the basis of the scope 1 and scope 2 GHG emission estimations, calculated in line with the applicable emission factors and predicted consumption values.
- Scope 1 emissions refer to direct GHG emissions that occur from sources that are owned or controlled by Kamoa. These are, for example, the emissions from fuels that are used on-site to operate machinery in the mining pit and on the surface around the processing complex.
- Scope 2 emissions refer to indirect GHG emissions from the generation of purchased electricity consumed by the Kamoa-Kakula mine. Even though most of the power grid of the DRC comes from hydropower, there is still a small fraction of electricity that is produced from thermal power plant. The GHG emissions from those thermal power plants are thus accounted for under Scope 2 emissions.

Data was obtained from Kamoa, the Project's Feasibility Study and other third party providers, this data was not independently verified by Hatch.

GHG emissions estimates (and the combined scope 1 and scope 2 GHG emission intensity metric) were calculated on an annual basis for each year from 2023–2038 (corresponding to the mine's anticipated full capacity production), based on the data provided and in accordance with the GHG Protocol Corporate Accounting and Reporting Standard (GHG Protocol, 2004). GHG emissions from the construction and closure phases of the Project were not included in this assessment, nor were scope 3 GHG data.

Various factors influence the GHG emission intensity that is associated with the production of a copper concentrate; these include ore head grade, mining method and geometry, mineralogical properties of the copper ore, ore hardness, and geographical factors. The Kamoa-Kakula Project's average GHG intensity metric over this period was estimated to be 0.16 tonnes of carbon dioxide equivalent per tonne of copper in concentrate produced (t CO_2 e/t Cu in concentrate) at the mine - based on estimated total annual average carbon dioxide equivalent (t CO_2 e/yr) emissions of 41,382 t CO_2 e/yr and an average annual copper in concentrate production of 271,288 t Cu/yr. Hatch has calculated a metric to two decimal places for comparative purposes only, and it should not be considered to be representative of the level of accuracy for the projected metric.

The limited benchmark study was based on publicly available information collected by Hatch. This public data was not independently verified by Hatch. The limited benchmark study indicated that the estimated GHG emission intensity for the Kakula mine and concentrator is among the lowest relative to other global operating mines and concentrators. Other mines powered with renewable energy sources were found to have a higher GHG emissions intensity ranging from 0.19 t CO_2 e/t Cu in concentrate to 2.8 t CO_2 e/t Cu in concentrate; and the only mine with a similar GHG intensity to that projected for the Kakula mine and concentrator is another underground mine in the DRC, which may also obtain its electricity from the national DRC energy grid and thereby benefit from the high proportion of hydroelectric power.

20.7.5 Assessment of Impacts

With the exception of the social impacts associated with physical and economic displacement (which remains of severe residual significance) and pending the results of the additional studies on critical habitats and Dilungus (Section 20.7.2), none of the other physical, biological or socio economic impacts are anticipated to be of high (severe or major) residual significance, following the effective implementation of the management actions as recommended in the ESMP (Golder, 2017 and CEMIC, 2019).

20.7.6 Management Actions and Monitoring Programme

A management actions and monitoring programme has been compiled as part of the EIA updates. (Golder, 2017 and CEMIC, 2019). The social and environmental management plan compiled by Kamoa covers all activities associated with early works. This plan needs to be updated to take the latest construction activities into consideration. It should also include measurable targets.

21 CAPITAL AND OPERATING COSTS

21.1 Cost Assumptions

Capital and operating costs for the Kakula 2020 FS and the Kakula-Kansoko 2020 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 Q1'20 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	0.88
CNY/USD	6.9
AUD/USD	1.43

21.2 Kakula 2020 FS

The Kakula 2020 FS cost model was prepared using current costs, existing contracts, quotations, labour rates, and other estimates. Unit costs were development and production quantities, labour numbers, and consumables estimates.

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 2020 FS Unit Operating Costs

	Payable Cu (US$/lb)		
	Years 1–5	**Years 1–10**	**LOM Average**
Mine Site	0.48	0.52	0.62
Transport	0.32	0.32	0.32
Treatment and Refining Charges	0.11	0.11	0.11
Royalties and Export Tax	0.20	0.20	0.20
Total Cash Costs	1.12	1.16	1.26

Table 21.3 Kakula 2020 FS Operating Costs

	Total LOM (US$M)	**Years 1–5**	**Years 1–10**	**LOM Average**
		(US$/t) Milled		
Site Operating Costs				
UG Mining	4,280	35.38	38.58	38.92
Processing	1,470	14.12	13.37	13.37
General and Administration	758	7.60	7.04	6.89
SNEL Discount	–294	–2.39	–2.55	–2.67
Customs Duties	245	2.13	2.21	2.23
Total	6,459	56.85	58.65	58.73

Table 21.4 Kakula 2020 FS Capital Cost Summary

Capital Costs (US$M)	Initial Capital (US$M)	Expansion Capital (US$M)	Sustaining Capital (US$M)	Total (US$M)
Underground Mining				
Underground Mining	131	202	538	871
Mining Infrastructure and Mobile Equipment	38	16	362	416
Capitalised Pre-Production	76	–	–	76
Subtotal	246	218	899	1,363
Off-site Power				
Power Supply Off Site	36	–	–	36
Subtotal	36	–	–	36
Concentrator and Tailings				
Plant	123	128	70	320
Tailings	13	26	88	127
Subtotal	136	154	157	448
Infrastructure				
Surface Infrastructure	69	101	14	184
Other Infrastructure	–	–	–	–
Contractor's and Owner's Camps	–	–	–	–
Subtotal	69	101	14	184
Indirects				
EPCM	35	17	0	53
Owners Cost	66	47	–	114
Customs Duties	8	18	40	66
Closure	–	–	82	82
Subtotal	110	83	122	315
Capital Expenditure Before Contingency	596	556	1,193	2,346
Contingency	50	38	72	159
Capital Expenditure After Contingency	646	594	1,265	2,505

21.2.1 Underground Mining Cost Estimates

This section describes the methods used to develop capital and operating estimates for the Kakula 2020 FS. The Kakula 2020 FS underground costs have been compiled by, DRA Projects (Pty) Ltd (DRA), Kamoa, Patterson & Cooke, SRK Consulting (SRK), Medyas Consulting (Backfill consultant) and Stantec (Mining). This section describes the cost estimate:

- Direct Mining – Stoping and development estimate.
- Backfill – Paste plant operation, UG distribution, backfill fence construction and paste fill.
- Underground Mining Infrastructure.

21.2.1.1 Assumptions and Key Estimating Criteria

The estimate is based on the following assumptions:

- Contractor rates as obtained from tenders as part of the Phase 1 construction work, were applied to all UG infrastructure costs.
- Preliminary and general (P&G) costs, applied to Contractor construction activities as obtained from exiting Phase 1 execution contracts and quotations received. P&G's include contractor supervision, rented equipment, consumables, accommodations, meals, transport, induction, training, wastages, etc.
- During the preproduction period, a mining contractor is used for lateral development and is limited to a six-crew maximum. A crew is defined as a development team with dedicated primary (jumbo and LHD) equipment. Contractors are also responsible for waste haulage activities. Development contractor will be involved until Q4'23.
- Contractors will complete all raise boring and boreholes for the life of the mine.
- Mobile equipment used by the Contractor will be transferred to owner once the contract is completed. This cost was captured as a depreciation cost on running hours and is priced towards the rebuild and replacement schedule of the equipment.
- Backfill fence construction, paste fill crews and paste reticulation UG will be done by owners' teams.
- Owner mining crews will comprise of local / Congolese labour only. This assumption is supported by the current training program enrolled by Kamoa and the training requirements of the mining contractor.
- Phasing out of expat labour was applied from Q1'23, post commissioning of the Phase 2 concentrator plant (Ramp-up to 7.6 Mtpa) and occurs over a year period where the local equivalent of the expat employee will be employed and is planned to work along with the expat for the year duration, until handover.

21.2.1.2 Allowances

Preliminary and General

A Preliminary and General (P&G) allowance has been added to underground infrastructure capital cost items by engineering discipline. This allowance was applied as a percentage to account for the following Contractor-provided activity or installation costs:

- Offices.
- Material Storage Areas.
- Workshops.
- Laboratories.
- Living Accommodation Off Site for Junior and Senior Staff.
- Transport of Workforce to and from Mine Site.
- Washing and Latrine Facilities.
- Tools and Equipment.
- Water Supplies, Electric Power, Communications.
- Access.
- Plant.
- Supervision for Duration of Construction.
- Contractor's Corporate Overheads and Profit (Mark-up).

Design Development Allowance

The design development allowance is to account for inaccuracies/scope growth in the estimate, either from designs, take-off quantities or rates used to provide the final estimate. The development allowance does not account for unforeseen circumstances, risk or omitted scope; hence it is not considered a contingency. Estimate input confidence levels of 5–25% were assigned depending on the proportion of the design that was complete.

21.2.1.3 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. A 10% contingency were applied against all capital cost items including mining and infrastructure.

21.2.1.4 Economic Base Date

All cost estimating is in January 2020 US Dollars (US$). A Limited amount of prices were used from June 2019, and these were escalated by 1% in U.S. Dollars terms prior to entry into the estimate.

21.2.1.5 Estimate Accuracy

The estimate was prepared to a feasibility study level of accuracy (from –10% to +15%). The first 18 months (2020 and 2021) are detailed monthly. The following four years are detailed quarterly (2021–2024). The remaining years of mine life are detailed on an annual basis.

21.2.1.6 Power Cost

The power rate applied to the Kakula 2020 FS operating cost model were based on actual invoices from August 2018 until August 2019 as provided by Kamoa. The Energy consumed in MWh per month vs. the maximum demand fluctuated between 58% and 79% which accounted for fluctuations on USD / kWh figures between USD 0.081 / kWh and USD 0.0652 / kWh. Once the mine enters a steady state operation the percentage of maximum demand will likely level out on 80%, hence a USD 0.065 / kWh rate was applied in the Kakula 2020 FS. Backup power will come from diesel generators with a cost of US$1.10/L diesel.

21.2.1.7 Labour

A total cost to company rate per Patterson grade as provided by Kamoa was applied to calculate labour cost. The total cost to company includes allowances for bonus, medical, employer contributions, two additional months salary, training, and agency costs. Nightshift and overtime allowances were calculated based on the current three shift rotation, 12-hour shift, 360 days per year cycle.

21.2.1.8 Mine development and stoping costs

Mining development and stoping costs are separated between contractor mining and owner mining costs.

Contractor Costs

Contractor mobilisation is a sunk cost and was not included in the Kakula 2020 FS. Contractor demobilisation is considered minimal, as the strategy is to retain equipment and facilities. Therefore, Contractor demobilisation is subject to a contract that is already in place and was not considered for this Study. Contractor activities are scheduled to end in Q4'23 and will reach a maximum of six crews. Contractor mining costs were applied as a USD/m based on the Contractors quoted rates. The contractor rate applied includes all direct and indirect costs associated with mine development, equipment depreciation, equipment running costs and contractor margin. Additional items not included by the contractor were added, such as cover drilling and permanent services (Electrical and permanent piping).

Owner Mining Costs

First principle-based mining costs were estimated for owner mining crews. A US$/m rate was estimated based on support requirements, excavation type (6 x 6, 7 x 6, room and pillar, drift and fill, mass excavations), single, double or multiple headings. This linked to the production metres produced the total costs for owner mining that includes the following:

- Direct labour.
- blasting equipment and consumables.
- Drilling consumables and equipment.
- Ground support - equipment and material.
- Services - dewatering pipes, service water pipes, EC&I, ventilation ducting.
- Mucking/hauling equipment.

21.2.1.9 Fixed and mobile equipment

Equipment capital and operating costs are split between mobile equipment and fixed equipment. Mobile equipment associated with mine development and stoping; fixed equipment is associate with UG rock handling, dewatering, ventilation, etc.

Mobile equipment quantities and operating hours were calculated based on the mining cycle times and production requirements. Mobile equipment capital costs were obtained from confirmed prices with equipment suppliers, currently supplying site. Equipment operating hours were used to estimate re-build and replacement frequencies over LOM. Operating costs includes fuel, tyres, parts and maintenance were all based vendor supplied Life Cycle Costing information and database consumption figures.

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.
- Replacement cost is assessed at 100% of the base unit cost plus options and freight.

Fixed equipment capital costs were obtained from suppliers pricing. Process Flow Diagrams produced a mechanical equipment list that was used to specify all the equipment required over LOM. Operating hours was estimated based on required throughput rates (t/hr or l/sec) from first principles. Operating costs were obtained mostly from suppliers with some database rates where supplier info was not available.

21.2.1.10 Backfill/paste fill

A backfill consultant provided cost estimate data for the backfill estimate. A first principle approach was applied to the backfill crews and equipment, based on productivities provided by the backfill consultant. Rates were assigned to equipment, labour and material. The scope of the backfill estimate include the following costs:

- Surface paste plant operation.
- Underground paste fill reticulation, including boreholes and installation of borehole pipes.
- Backfill fences.
- Sand fill.
- Paste fill.

All backfill costs included in the Kakula 2020 FS assume owner crew installation and operation of all aspects of paste fill except the installation of the pipes in the boreholes, which is supplied and installed by a specialist contactor.

Underground Capital Costs

The underground capital costs were estimated for the following:

- Surface Materials Handling Facilities with Boreholes (explosives, fuel and lube, Backfill and concrete/shotcrete).
- Electrical, Control, Communications, and Instrumentation Systems.
- Main Workshops, satellite workshops with Offices and Stores.
- Underground Materials Handling Facilities (explosives, fuel and lube, concrete/shotcrete).
- Truck tips, feeders and conveyor belts.
- Piping Services and Water Handling.
- Dewatering System.
- Ventilation Raises, Fans, Controls.
- Mine Air Refrigeration.

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.

Direct and Indirect Costs

Capital and operating costs were subdivided into direct and indirect costs, based on the Kakula 2020 FS work breakdown structure (WBS) or the Kamoa Copper SA 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 labour.
- 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 Clean-up.
- 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).

Engineering, Procurement, and Construction Management Allowance

The current Phase 1 EPCM labour forecast and a first principle estimate for the Phase 2 EPCM formed the basis for Plant, Mining and Surface infrastructure EPCM costs, provided by DRA.

EPCM included for mining is for underground infrastructure and assistance to the site team to purchase mobile equipment, raise boring, borehole drilling and underground services. The site execution team are responsible for the mining contractor and owner mining, and hence the cost is excluded from EPCM. A separate EPCM cost for the TSF was provided by the TSF consultant.

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 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 labour, daily expenditures, and all equipment operating costs. The current and proposed Kakula-specific labour, which would perform the duties of the Owner's Team, are included in the Kakula Indirects. All non-labour 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 based on equipment operating hours 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%.
- Generated Power Cost: $0.45 KWhr (Diesel consumption, oil and maintenance costs).
- Grid Overland Power Cost: $0.065 KWhr (2020–2041).

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:

- Blasting authorisation – US$150 per day.
- Traceability attestation – 2% of explosive cost.
- Authorisation of purchase, transport, and storage – US$1,000 every three months.
- Blasting tickets – US$1,800 /blaster that includes US$1,500 for training and US$300 tax applied per annum.
- Blasting assistance – An allowance of US$8,400 per month were applied to account for assistants that will be employed by the mine.

21.2.2 Concentrator and Site Infrastructure Costs

This section describes the basis and methodology used to prepare the capital cost estimate 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, expansion capital and sustaining costs for the life of mine. The Kakula concentrator design is based on a phased approach of two 3.8 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 concentrator and site infrastructure costs in the capital cost estimate includes the Kakula 2020 FS scope as discussed in Section 17 and Section 18.

General

The Kakula capital cost estimate meets the required accuracy criteria of –10% +15% and complies with a Class 1 FS 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 January 2020 US$.

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 (EC&I) supply and erection.
- Infrastructure buildings.
- Transportation to site.

- EPCM services.
- First fills of consumables.
- Spares.

Bulk Earthworks and Infrastructure

The large bulk earthworks for were quantified from modelled quantities that are being used for the earthworks tender. Earthworks rates were provided by the selected earthworks contractor who was appointed to execute the work.

Geotechnical

The following estimating 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

Internal Roads are gravel and dirt roads are constructed during Phase 1 and then upgraded to paved roads during Phase 2 in an attempt to defer capital costs. All roads not in the mining and plant areas will remain gravel roads over LOM.

External roads including the main access road (airport road) is currently being constructed by Kin Baton under Mariswe's site supervision. The forecasted construction completion cost has been used for the estimate.

A new concentrate export road (Kotonoto export road) will be constructed as part of Phase 2, which will link the process plant with the east–west main road to Angola. The road is built to the same specification as the main access road.

Buried Services and Storm Water Reticulation

Measurements off the block plan were used to quantify the distances of buried services and number of culverts required. The civil contractor will supply and install the sewage lines. As for the rest of the piping, the SMPP contractor will supply and install the pipes while the civil contractor will dig and close the necessary trenches.

Fencing

Boundary fences, such as the 16 km fence and the 3 km fence, are sunk costs and in this estimate an allowance was made for additional internal fencing with access control gates. An allowance in Phase 2 was made to upgrade the 16 km fence.

Pollution Control Dam

A 15 Ml process water dam is to be constructed during Phase 1, sized according to the water requirements during the LOM.

Storm water drains are included in the earthworks budget, as part of the terraces. All storm water dams were moved to the 2nd phase (including silt traps).

The following assumptions were made for estimating purposes:

- 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 Kakula 2020 FS. 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 85–110 kg/m^3.
- The BOQ was then populated with rates received from RLB Pentad.

Infrastructure Building Works

Infrastructure and building layouts were measured from drawings approved by Kamoa. Requests for quotations were issued to the market and rates were used to populate a rate per square metre.

Three building types are used for different purposes in the mine:

- Modular buildings – Change house, lamp room, offices, etc.
- Light steel buildings with sheeted walls and civil bases.
- Electrical buildings constructed of steel, concrete and brick.

Estimates were then populated based on the approved drawings and applicable areas.

Structural Steelwork and Platework Supply and Erection

Steelwork quantities for all structures were estimated from general arrangement and layout drawings produced by DRA. Steel will be Chinese supply and install. Chinese steel supply rates are based on South African grade (355) type steel. A 5% increase in weight was allowed to account for Chinese member sizes which may not match standard South African member sizes and consequently a larger size may need to be selected. Platework is excluded from this allowance. The SMPP contractor also included construction of conveyors; however, conveyor steel supply is from a South African supplier, currently supplying steel to site as part of the Phase 1 construction.

A detailed BOQ was provided by a Chinese construction company which contained erection costs for all the mechanical items in the estimate, including conveyor mechanicals. These rates were applied to the final quantities as estimated in the Kakula 2020 FS.

The SMP value includes shop detailing, corrosion protection, supply, transportation to site, off-loading and erection including all their associated costs with regards to induction, medicals, accommodation and meals, personal protective equipment (PPE), personal protective clothing (PPC), travelling, etc. to complete the works in full.

Mechanical Equipment

Using the MEL as a basis, DRA issued enquiries for all major mechanical equipment to vendors for costing as part of the Basic Engineering Phase 1 (Phase 1 construction), requesting budget quotations. Mechanical equipment datasheets for all major equipment were approved by the client prior to enquiry. Enquiries were then issued to vendors for costing. Pricing has been updated for the Kakula 2020 FS on items to align pricing with 2020 figures. The rates used for the Kakula 2020 FS update were obtained from placed contracts.

The erection cost for the mechanical equipment was included in the SMPP RFQ, as described under steelwork.

For minor and/or ancillary mechanical equipment items, supply costs were obtained from previous quotations and/or the DRA historical database; however, this was applied to a very small part of the estimate.

Belt Conveyors

Designs of all belt conveyors were carried out by the DRA engineering department in accordance with the belt profiles as depicted on the general arrangement drawings, including the need to meet the process requirements and 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 costs from recent projects stored in 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 were assigned to all transformers and motor control centres (MCCs) for all areas using the WBS and MEL. The MEL is used to estimate/calculate load centres and consequently size the required electrical equipment. Starter panel types (DOL/VSD, etc.) were assigned as per the MEL. EC&I rates and designs were based on the latest designs and contract rates for supply and erect that is currently underway as part of the Phase 1 EPCM work (Phase 1).

The following basis was used for the items specified below:

- Cables: A detailed cable schedule was carried out to determine the size and quantities of cables. An average length of 150 m has been allowed per drive/motor and a run of 50 m per MCC. MV and overland cables were measured as per the SLD from the block plans and mining design. Cables for underground mining installations are based on Phase 1 designs for typical areas.
- Cable racking: Based on quantities as per the detailed designs for Phase 1 concentrator plant and the backfill. Quantities for underground installations are based on present designs for certain areas and the 3D mining model.
- Field isolators: Quantities as per the load list and rates as per Phase 1 contract.
- MCC and starters: Quantities as per the load list and rates as per Phase 1 contract.
- Transformers: Quantities as per the SLDs. Rates as per the Phase 1 contract.
- Power factor correction (PFC): PFC required is as per the load list and preliminary design. The cost is as per DRA database cost per MVAr.
- HT switchgear: 33/11 kV quantities as per the single line diagram. Cost is as per the Phase 1 contract.

- 220 kV switchgear was costed with the aid of the single line diagram according to information received from suppliers.
- Lighting and small power: No detailed engineering involved, costed on a per square metre basis for each area.
- Installation cost: Rates as per the Phase 1 contract were used.

Infrastructure Loads:

- Mini substation was assigned to grouped loads. Costs as per Phase 1 contract.
- DRA database costs were applied for low voltage DBs.
- Buildings, workshops and offices: DRA database values were applied.
- 11kV overhead lines: Lengths measured off block plans and Phase 1 contract rates applied.
- Conveyor belt lighting: Costed from the Phase 1 rates and quantities measured from layout drawings.
- Earthing and lightning protection: Provisional quantities and rates as per Phase 1 contract. DRA database values were applied.

Control and Instrumentation costs included in the Phase 1 estimate were based on the Phase 1 BE estimate where these costs were factorised on MV. A 35% factor was applied to MV costs to estimate C&I. this is only limited to the Phase 1 portion of the estimate. The factor is based on costs from similar DRA projects.

Phase 2 control and instrumentation estimates are based on a detailed instrumentation take-off from the P&IDs. The control network diagrams were updated to include the requirements for concentrator Phase 2 and backfill plant Phase 2. Quantities for the control equipment were determined from the control network diagrams. Costs for the C&I equipment are as per Phase 1 contracts.

A total of 16 x 1.8 MW Emergency Power from Sumec generator sets will be installed. The rates are based on a recent contract that was placed on these generatorsTransportation.

Transportation costs for steelwork and platework were quoted in the SMPP rates. Other transport costs were based on a tonnage or “per load” basis. Rates from the logistic service provider were used. 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 storage and distribution system.
- Sewerage treatment plant.
- Potable water treatment plant.
- Waste-water treatment plant.
- Laboratory equipment.

Spares

Allowances have been made for the first fill of oil and lubrication, reagents and grinding media.

The spares holding costs have been derived from vendor recommendations, as per quotations received. The spare parts costs have been grouped per item of equipment in a separate section within the estimate. Where no spares were quoted by the vendor, and it has been deemed by DRA that there should be a spare holding, a percentage of the supply price has been applied.

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.
- EC&I – 5.0% of EC&I supply.
- Valves – 3.0% of piping and valves supply.

Operational Spares:

- Two-year operational spares were excluded from the CBE. The costs are accounted for under the Opex estimate in the form of normal running costs.

Construction Facilities Costs

Allowances have been made in the capital cost estimate 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.
- Commissioning tools.

21.2.3 Owner's Cost and G&A

Kamoa Copper SA have prepared a budget for Owners costs these were reviewed and adjusted to allow for capital and operating costs for the life of mine estimate. 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.
- Security and Protection Services.
- Medical Support.
- Expatriate Flights.
- Light Vehicles.
- Environmental.
- Community Development.
- Banking and Audit Fees.
- Legal and Consultants.
- Studies.
- Resettlement.
- Capitalised General and Administration costs.

21.3 Kakula-Kansoko 2020 PFS

Kakula-Kansoko 2020 PFS comprises the Kakula 2020 FS (6 Mtpa) and additional 1.6 Mtpa produced from Kansoko. The Kakula 2020 FS Capital infrastructure remains the same as described above with the summary tables below indicating the combined capital costs of Kakula-Kansoko 2020 PFS (7.6 Mtpa).

The Kakula-Kansoko 2020 PFS cost model was prepared using current costs, existing contracts, quotations, labour rates, and other estimates. Unit costs were development and production quantities, labour numbers, and consumables estimates.

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 Kakula-Kansoko 2020 PFS Unit Operating Costs

	Payable Cu (US$/lb)		
	Years 1–5	**Years 1–10**	**LOM Average**
Mine Site	0.50	0.55	0.64
Transport	0.35	0.35	0.42
Treatment and Refining Charges	0.12	0.12	0.13
Royalties and Export Tax	0.21	0.22	0.25
Total Cash Costs	1.18	1.23	1.44

Table 21.6 Kakula-Kansoko 2020 PFS Operating Costs

	Total LOM (US$M)	Years 1–5	Years 1–10	LOM Average
		(US$/t) Milled		
Site Operating Costs				
UG Mining	8,134	35.90	38.45	34.59
Processing	3,143	13.72	13.20	13.37
Tailings	45	0.21	0.18	0.19
General and Administration	1,198	7.61	7.19	5.09
SNEL Discount	–545	–2.45	–2.59	–2.32
Customs Duties	476	2.14	2.21	2.02
Total	12,451	57.14	58.64	52.95

Table 21.7 Kakula-Kansoko 2020 PFS Capital Cost Summary

Capital Costs (US$M)	Initial Capital (US$M)	Expansion Capital (US$M)		Sustaining Capital (US$M)	Total (US$M)
		Kakula 6.0 Mtpa / 7.6 Mtpa Plant	Kansoko to 1.6 Mtpa		
Underground Mining					
Underground Mining	158	202	97	1,068	1,525
Capitalised Pre-Production	76	–	–	–	76
Mining Mobile Equipment	55	43	17	922	1,036
Subtotal	289	245	114	1,990	2,638
Off-site Power					
Power Supply Off Site	36	–	–	–	36
Subtotal	36	–	–	–	36
Concentrator and Tailings					
Plant	123	128	–	135	386
Tailings	13	12	–	240	265
Subtotal	136	139	–	375	651
Infrastructure					
Plant Infrastructure	69	101	–	14	184
Conveyor Kansoko–Kakula	–	–	–	95	95
Subtotal	69	101	–	109	279
Indirects					
EPCM	37	15	9	0	62
Owners Cost	67	47	4	–	117
Customs Duties	8	23	–0	89	120
Closure	–	–	–	81	81
Subtotal	113	85	13	170	380
Capital Expenditure Before Contingency	642	571	126	2,644	3,984
Contingency	52	41	12	183	288
Capital Expenditure After Contingency	695	612	139	2,827	4,272

21.3.1 Underground Mining Cost Estimates

This section describes the parameters and the capital and operating cost basis of estimates to support the Kakula-Kansoko 2020 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 Q2'20 US$. The Kakula-Kansoko 2020 PFS uses the costs from the Kakula 2020 FS and costs prepared for Kansoko to PFS level of accuracy. All Kakula costs are that of the Kakula 2020 FS described above and prepared by DRA. All Kansoko costs have been prepared by OreWin.

Underground Capital Costs

The total capital cost includes both initial, expansion and sustaining capital. Initial 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:

- 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, controlled convergence room-and-pillar, 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, Controlled Convergence 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.

Contractor Profit, Overhead, and Allowances

For Contractor development and excavation, the Stantec estimate includes an 18% profit and overhead (mark-up) applied to equipment, materials, and labour 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:

- Metres (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 (kg) for weight (e.g. explosives, fabricated steel).
- Litres per second (L/s) for flowrate (e.g. pumping).
- Metres 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 all costs from the start of Q3'20 until mill start-up at the end of Q2'21, all costs prior to this are considered sunk. Starting from 1 July 2020. Existing on site contractor equipment is available for the Study period, therefore, capital equipment was reduced accordingly. The Initial preproduction period is 12 months, and the ends upon mill start up, end of Q2'21. The production period is from the end of preproduction through the end of LOM.

Project cost estimates are broken into the following four main classifications (Initial, expansion, sustaining capital, and operation) and two cost types (direct and indirect).

Initial Preproduction Capital

Initial 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.
- 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 Authorisations 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 labour, 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 organisation of the expenditure schedule.

Direct costs are cash costs directly associated with the output of a unit of production (i.e., per metre 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 labour.
- 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 Clean-up.
- 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%.

Labour

The Contractor's labour 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 provided monthly labour rates for local and expat labour. 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 day/month and 12 h/day to obtain an hourly rate for estimating the cost.

Contractor and Owner local labour rates are the same. Contractor expatriates were given a 42% hourly increase over Owner expatriates.

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 mark-up for contractor-owned equipment used to complete all tasks from January 2019 to Q4'24. Equipment rentals are charged at a monthly rate, which is 3.25% of total replacement value plus mark-up for this Study. The rental rate covers the contract strategy of no cost being incurred to transfer ownership to the Owner in Q4'24.

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 (mark-up) 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. The following key elements that were used to develop the unit cost database are:

- 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

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 labour, daily expenditures, and all equipment operating costs. The current and proposed Kansoko-specific labour, which would perform the duties of the Owner's Team, are included in the Kansoko Indirects. All non-labour costs associated with the Owner's Team are included in the Kansoko.

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 (2020).
- Grid Overland Power Cost: $0.065 KWhr (2021–2057).

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. 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 two years of development for development equipment and no replacement charges for other equipment in the final two 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 labour, 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.

21.3.2 Concentrator and Site Infrastructure Costs

This section describes the basis and methodology used to prepare the capital cost estimate for the Kakula concentrator together with site infrastructure capital costs, project indirect costs, infrastructure costs at Kakula and Kansoko, and contingencies for the initial capital requirements for the execution of the Kakula and Kansoko project phase, expansion capital and sustaining costs for the life of mine.

The Kakula concentrator design is based on a phased approach of two 3.8 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 concentrator and site infrastructure costs in the capital cost estimate includes the PFS scope as discussed in Section 17 and Section 18.

General

The Kakula capital cost estimate meets the required accuracy criteria of –10% +15% and complies with a Class 1 FS 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 January 2020 US$.

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 (EC&I) supply and erection.
- Infrastructure buildings.
- Transportation to site.
- EPCM services.
- First fills of consumables.
- Spares.

Kansoko PFS Conveyor

A PFS design was done on the 6.0 Mtpa conveyor that will be installed to convey crushed rock from Kansoko mine to the Kakula concentrator complex once Kakula mining is completed. The conveyor is approximately 11 km long and is designed for daytime running only at a capacity of 1,420 t/hr.

The estimate was compiled per discipline as follow:

- Earthworks and civils – The estimate was completed based on long sections that was used to calculate cut and fill quantities for earthworks. The number of road crossings and culvert crossings were determined from the conveyor route. Typical culverts and road crossings were used. Rates used were based on the current rates used by the contractors working on site.

- Steelwork, platework and piping – quantities were applied on a per metre basis, based on a typical cross section of the conveyor. Platework for head and tail-end was done per installation. Rates from the current site construction work was used.
- Mechanical – Mechanical equipment requirements were based on the functional design report completed by DRA on the conveyor. Conveyor lengths, lift heights and capacity are factors considered to estimate the required drives, pulleys, bending radius etc. Rates for mechanical kit was obtained from current suppliers and database rates were used where required. Erect costs were obtained from database rates.
- EC&I – a single line diagram was compiled based on the Mechanical Equipment list that was provided. The SLD was used to estimate the electrical costs, considered current rates as received from suppliers and contractors currently used in the phase 1 construction.

Bulk Earthworks and Infrastructure

The large bulk earthworks for were quantified from modelled quantities that are being used for the earthworks tender. Earthworks rates were provided by the selected earthworks contractor who was appointed to execute the work.

Geotechnical

The following estimating 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

Internal Roads are gravel and dirt roads are constructed during Phase 1 and then upgraded to paved roads during Phase 2 in an attempt to defer capital costs. All roads not in the mining and plant areas will remain gravel roads over LOM.

External roads including the main access road (airport road) is currently being constructed by Kin Baton under Mariswe's site supervision. The forecasted construction completion cost has been used for the estimate.

A new concentrate export road (Kotonoto export road) will be constructed as part of Phase 2, which will link the process plant with the east–west main road to Angola. The road is built to the same specification as the main access road.

Buried Services and Storm Water Reticulation

Measurements off the block plan were used to quantify the distances of buried services and number of culverts required. The civil contractor will supply and install the sewage lines. As for the rest of the piping, the SMPP contractor will supply and install the pipes while the civil contractor will dig and close the necessary trenches.

Fencing

Boundary fences, such as the 16 km fence and the 3 km fence, are sunk costs and in this estimate an allowance was made for additional internal fencing with access control gates. An allowance in Phase 2 was made to upgrade the 16 km fence.

Pollution Control Dam

A 15 Ml process water dam is to be constructed during Phase 1, sized according to the water requirements during the LOM.

Storm water drains are included in the earthworks budget, as part of the terraces. All storm water dams were moved to the 2nd phase (including silt traps).

The following assumptions were made for estimating purposes:

- An allowance has been made for fill and 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 Kakula-Kansoko 2020 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 85–110 kg/m^3.
- The BOQ was then populated with rates received from RLB Pentad.

Infrastructure Building Works

Infrastructure and building layouts were measured from drawings approved by Kamoa. Requests for quotations were issued to the market and rates were used to populate a rate per square metre.

Three building types are used for different purposes in the mine:

- Modular buildings – Change house, lamp room, offices, etc.
- Light steel buildings with sheeted walls and civil bases.
- Electrical buildings constructed of steel, concrete and brick.

Estimates were then populated based on the approved drawings and applicable areas.

Structural Steelwork and Platework Supply and Erection

Steelwork quantities for all structures were estimated from general arrangement and layout drawings produced by DRA. Steel will be Chinese supply and install. Chinese steel supply rates are based on South African grade (355) type steel. A 5% increase in weight was allowed to account for Chinese member sizes which may not match standard South African member sizes and consequently a larger size may need to be selected. Platework is excluded from this allowance. The SMPP contractor also included construction of conveyors; however, conveyor steel supply is from a South African supplier, currently supplying steel to site as part of the Phase 1 construction.

A detailed BOQ was provided by a Chinese construction company which contained erection costs for all the mechanical items in the estimate, including conveyor mechanicals. These rates were applied to the final quantities as estimated in the FS.

The SMPP value includes shop detailing, corrosion protection, supply, transportation to site, off-loading and erection including all their associated costs with regards to induction, medicals, accommodation and meals, personal protective equipment (PPE), personal protective clothing (PPC), travelling, etc. to complete the works in full.

Mechanical Equipment

Using the MEL as a basis, DRA issued enquiries for all major mechanical equipment to vendors for costing as part of the Basic Engineering Phase 1 (Phase 1 construction), requesting budget quotations. Mechanical equipment datasheets for all major equipment were approved by the client prior to enquiry. Enquiries were then issued to vendors for costing. Pricing has been updated for the Kakula-Kansoko 2020 PFS on items to align pricing with 2020 figures. The rates used for the Kakula-Kansoko 2020 PFS update were obtained from placed contracts.

The erection cost for the mechanical equipment was included in the SMPP RFQ, as described under steelwork.

For minor and/or ancillary mechanical equipment items, supply costs were obtained from previous quotations and/or the DRA historical database; however, this was applied to a very small part of the estimate.

Belt Conveyors

Designs of all belt conveyors were carried out by the DRA engineering department in accordance with the belt profiles as depicted on the general arrangement drawings, including the need to meet the process requirements and 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 costs from recent projects stored in 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 were assigned to all transformers and motor control centres (MCCs) for all areas using the WBS and MEL. The MEL is used to estimate / calculate load centres and consequently size the required electrical equipment. Starter panel types (DOL/VSD, etc.) were assigned as per the MEL. EC&I rates and designs were based on the latest designs and contract rates for supply and erect that is currently underway as part of the Phase 1 EPCM work (Phase 1).

The following basis was used for the items specified below:

- Cables: A detailed cable schedule was carried out to determine the size and quantities of cables. An average length of 150 m has been allowed per drive/motor and a run of 50 m per MCC. MV and overland cables were measured as per the SLD from the block plans and mining design. Cables for underground mining installations are based on Phase 1 designs for typical areas.
- Cable racking: Based on quantities as per the detailed designs for Phase 1 concentrator plant and the backfill. Quantities for underground installations are based on present designs for certain areas and the 3D mining model.
- Field isolators: Quantities as per the load list and rates as per Phase 1 contract.
- MCC and starters: Quantities as per the load list and rates as per Phase 1 contract.
- Transformers: Quantities as per the SLDs. Rates as per the Phase 1 contract.

- Power factor correction (PFC): PFC required is as per the load list and preliminary design. The cost is as per DRA database cost per MVAr.
- HT switchgear: 33/11 kV quantities as per the single line diagram. Cost is as per the Phase 1 contract.
- 220 kV switchgear was costed with the aid of the single line diagram according to information received from suppliers.
- Lighting and small power: No detailed engineering involved, costed on a per square metre basis for each area.
- Installation cost: Rates as per the Phase 1 contract were used.

Infrastructure Loads:

- Mini substation was assigned to grouped loads. Costs as per Phase 1 contract.
- DRA database costs were applied for low voltage distribution boards.
- Buildings, workshops and offices: DRA database values were applied.
- 11 kV overhead lines: Lengths measured off block plans and Phase 1 contract rates applied.
- Conveyor belt lighting: Costed from the Phase 1 rates and quantities measured from layout drawings.
- Earthing and lightning protection: Provisional quantities and rates as per Phase 1 contract. DRA database values were applied.

Control and Instrumentation costs included in the Phase 1 estimate were based on the Phase 1 BE estimate where these costs were factorised on MV. A 35% factor was applied to MV costs to estimate C&I. this is only limited to the Phase 1 portion of the estimate. The factor is based on costs from similar DRA projects.

Phase 2 control and instrumentation estimates are based on a detailed instrumentation take-off from the P&IDs. The control network diagrams were updated to include the requirements for concentrator Phase 2 and backfill plant Phase 2. Quantities for the control equipment were determined from the control network diagrams. Costs for the C&I equipment are as per Phase 1 contracts.

A total of 16 x 1.8 MW Emergency Power from Sumec generator sets will be installed. The rates are based on a recent contract that was placed on these generators.

Transportation

Transportation costs for steelwork and platework were quoted in the SMPP rates. Other transport costs were based on a tonnage or "per load" basis. Rates from the logistic service provider were used. 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 storage and distribution system.
- Sewerage treatment plant.
- Potable water treatment plant.
- Waste-water treatment plant.
- Laboratory equipment.

Spares

Allowances have been made for the first fill of oil and lubrication, reagents and grinding media.

The spares holding costs have been derived from vendor recommendations, as per quotations received. The spare parts costs have been grouped per item of equipment in a separate section within the estimate. Where no spares were quoted by the vendor, and it has been deemed by DRA that there should be a spare holding, a percentage of the supply price has been applied.

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.
- EC&I – 5.0% of EC&I supply.
- Valves – 3.0% of piping and valves supply.

Operational Spares:

- Two-year operational spares were excluded from the CBE. The costs are accounted for under the Opex estimate in the form of normal running costs.

Construction Facilities Costs

Allowances have been made in the capital cost estimate 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.
- Commissioning tools.

Owner's Cost and G&A

Kamoa Copper SA have prepared a budget for Owners costs these were reviewed and adjusted to allow for capital and operating costs for the life of mine estimate. 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.
- Security and Protection Services.
- Medical Support.
- Expatriate Flights.
- Light Vehicles.
- Environmental.
- Community Development.
- Banking and Audit Fees.
- Legal and Consultants.
- Studies.
- Resettlement.
- Capitalised General and Administration costs.

21.4 Comments on Section 21

In the opinion of the QPs, the work completed for the Kakula 2020 FS and Kakula-Kansoko 2020 PFS adequately support the studies. There is high confidence in rates used in the estimate due to the current ongoing construction work on the estimating quantities, which are based on layout drawings that are deemed adequate for the level of study.

22 ECONOMIC ANALYSIS

Two levels of economic analysis were undertaken for the Project, assuming mining of the Mineral Reserve utilising the processing rate defined in the Kakula 2020 FS and in the Kakula-Kansoko 2020 PFS. Both studies support the Mineral Reserve.

22.1 Economic Assumptions

The modelling and taxation assumptions used in the Kakula 2020 FS and the Kakula-Kansoko 2020 PFS is discussed in detail below.

The studies have been prepared on the assumption that the 2018 DRC Mining Code applies to the project, including the super profits tax that was introduced by that code in 2018.

As Ivanhoe Mines originally disclosed in March 2018, it and other mining industry participants had expressed concerns to the DRC government regarding the implementation of the 2018 Mining Code. In particular, Ivanhoe Mines sought, and continues to seek, assurances from the DRC government that it will honour the clear guarantee of stability contained in Article 276 of the former 2002 Mining Code. The stability guarantee states as a matter of law that holders of DRC exploration and exploitation permits would continue to benefit from rights granted under the 2002 Mining Code “for a period of 10 years” after the implementation of any legislated amendment, which includes the 2018 Mining Code.

Ivanhoe Mines' investments in the DRC were made on the basis that it would have the benefit of the stability clause, and as a result the Kamoa-Kakula Project would not be exposed to changes, including the super profits tax, for 10 years following any legislative change. In particular, the stability clause permitted an investment decision to be justified at copper prices less than US$3.00/lb, whereas with the 2018 Mining Code changes, a higher copper price, indexed to inflation, would be needed to offset the 2018 changes and loss of the protection of the Article 276 stability clause. Ivanhoe considers a nominal, inflated copper price above US$3.10/lb. as the basis for determining the super profits tax, should it ultimately be unsuccessful in securing the continued benefit of the stability clause.

Submissions have been made by Ivanhoe Mines and other mining industry participants which seek to address and resolve the stability arrangements and other items of concern with the 2018 Mining Code. While meetings have been held among Ivanhoe Mines, mining industry representatives and members of the current and former DRC governments, including the President, the concerns surrounding the 2018 Mining Code have not been resolved. Further discussions have been delayed, in part, due to the ongoing COVID-19 pandemic, and accordingly, these studies assume the applicability of the 2018 Mining Code. However, once these issues are resolved, Ivanhoe Mines may revise the results of these studies to take that resolution into account, which Ivanhoe Mines expects would include the continued applicability of the stability clause of the 2002 Mining Code.

22.1.1.1 Pricing and Discount Rate Assumptions

The key assumptions in the economic modelling relating to product pricing are tabulated in Table 22.1. A discount rate of 8% is used for calculating net present value.

Table 22.1 Kakula 2020 FS Capital Cost Summary

Model Assumption	Value
Copper price (US$/lb)	3.10
Concentrate treatment charge (US$/t concentrate)	62
Concentrate refining charge (US$/lb Cu)	0.062
Concentrate Transport to China (US$/t concentrate)	364
Copper pay ability (%)	96.75
DRC Provincial Tax on concentrate (US$/t concentrate)	100
Community Development Contribution (% total revenue)	0.30
DRC Mining Royalty (%)	3.50
DRC Export Taxes (%)– inc. "Redevance Informatique"	1.25

The Project level valuation model begins on 1 July 2020. It is presented in Q2'20 constant dollars; cash flows are assumed to occur evenly during each year and a mid-year discounting approach is taken.

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 and payables is assumed to be four weeks and six weeks on average.

22.1.1.2 Taxation

In the DRC, companies that are holders of mining rights are subject to 30% taxation on net income. The economic model applies this taxation rate after accounting for operating costs and depreciation on capital investments.

Provincial taxation on copper concentrate and national export tax is applied in the economic model on copper concentrate production. These taxes are applied independently of capital and operating costs.

22.1.1.3 Royalties

The royalty is due upon the sale of the product and is calculated at 3.5% of the gross commercial value of non-ferrous metals.

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

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 corporate income tax (CIT) based on tax at 30% on net income, excluding transportation costs.

A minimum tax of 1% of revenue, which is deductible from the CIT basis, is payable in the event that a mining company is in a loss position or offsetting previous losses carried forwards (see below), and withholding tax on distributions are subject to 10% tax at the shareholder's level. In addition, as from 1 January 2014, the minimum amount of tax payable by mining companies in a year is 1% of the calculated revenue for that specific year.

Tax Holidays

The DRC tax legislation does not currently provide for any tax holiday incentives.

Tax Losses

Tax losses from a financial year may be deducted from profits earned in subsequent years up to the fifth year following the loss-making period. The aggregate exploration expenditure may be claimed.

Research and Development Costs

Research and development costs capitalised during the exploration and construction phases may be amortised over a period of two years from first production, with losses resulting from such an amortisation allowed to be carried forwards. These costs include exploration, owners' costs, certain underground development costs, and interest paid on shareholders loans.

Depreciation

Specific mining assets dedicated to mining operations, with useful lives between 4–20 years are depreciated on a straight line basis.

Non-mining assets are depreciated in accordance with the common law. The common low provides different depreciation rates for various assets, e.g. 10-years for plant and equipment.

Value Added Tax

Value added tax (VAT) came into effect in the DRC in January 2012. VAT is levied on all supplies of goods and services at a rate of 16% and is not levied on any capital asset movements.

VAT Exoneration

Holders of mining rights are currently entitled to exoneration for certain import duties and import VAT for materials and equipment imported for construction of a mine and related infrastructure in accordance with the VAT Act, subject to any unforeseen changes in the law. Kamoa Copper SA has successfully received the exoneration in the past in terms of the mining code and it expects to receive such exoneration for most imports for project construction in future in compliance with the mining code.

Customs / Import Duties

Imports of equipment are subject to an entry fee at the rate of:

- 2% before the beginning of the mining operations.
- 5% until the end of the third year of the first production.

All intermediate goods and other consumables are taxed at the rate of 10% of tariffs, except fuels and lubricants for mining activities which are subject to the rate of 5%. The common law system is applicable to all imports after five years from the date of the granting of the mining licence.

Export Taxes

Fees and royalties for services rendered – claimed by DRC Agencies and specific public services – is limited to 1.0% of the gross commercial value of the export. In addition, a levy of 0.25% of the gross commercial value is charged for Redevance Informatique.

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 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 this 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.

22.1.1.5 Sunk Costs

The estimate excludes all sunk costs up to 30 June 2020.

22.2 Kakula 2020 FS Overview and Results

Kamoa Copper SA is currently developing the Kakula Mine. The base case described in the Kakula 2020 FS is the construction and operation of an underground mine, concentrator processing facilities, and associated infrastructure.

The Kakula Project is planned to mine and process 6.0 Mtpa of ore production over a production period of 21-years. The LOM production scenario provides for 110.0 Mt to be mined at an average grade of 5.22% copper, producing 8.5 Mt of high-grade copper concentrate, containing approximately 10.8 billion pounds of copper.

The Kakula 2020 FS evaluates the development of a 6.0 Mtpa underground mine and surface processing complex based on mining the Kakula deposit. The mill would be constructed in two phases of 3.8 Mtpa each as the mining operations ramp-up to full production of 6.0 Mtpa, which leaves spare capacity in the mill of 1.6 Mtpa. The first module of 3.8 Mtpa commences production in Q3'21, and the second in Q1'23. This staged development scenario for Kakula is shown in Figure 22.1.

Figure 22.1 Kakula 2020 FS 6.0 Mtpa Development Scenario

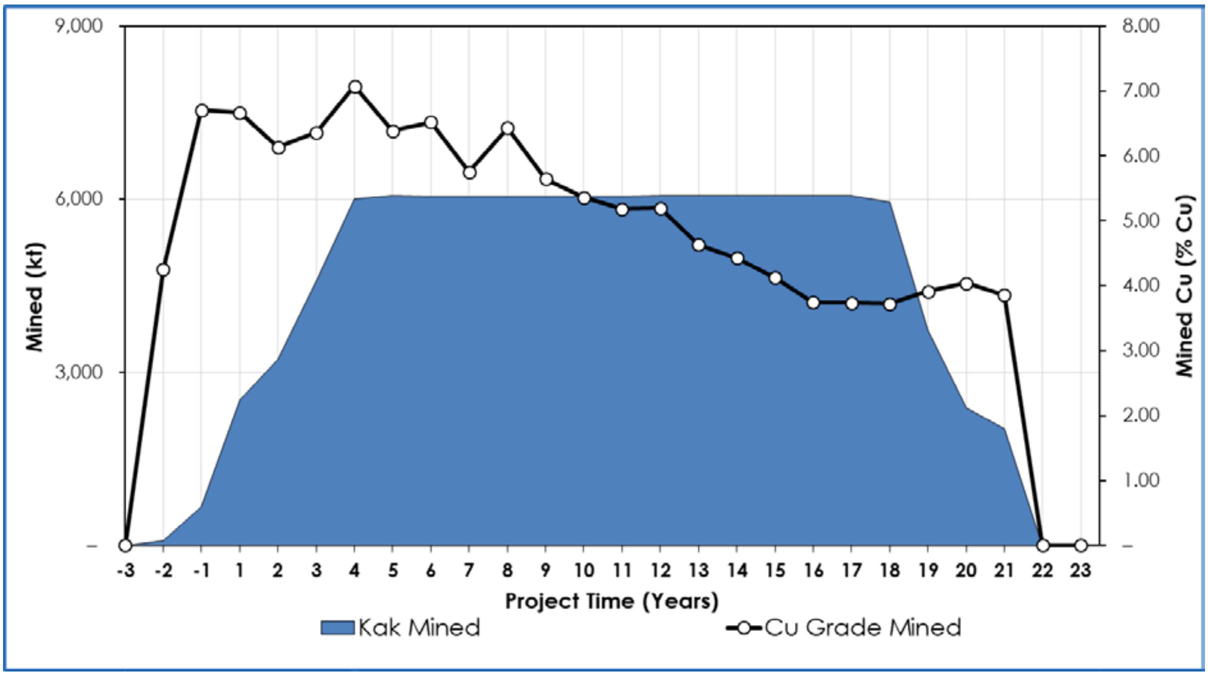


Figure by OreWin, 2020.

22.2.1 Summary of Key Physical and Financial Metrics - Kakula

A summary of the key results for the Kakula 2020 FS scenario are:

- Very-high-grade initial phase of production is projected to have a grade of 7.1% copper in Year-4 and an average grade of 6.2% copper over the initial 10-years of operations, resulting in estimated average annual copper production of 284,000 tonnes.
- Annual copper production is estimated at 366,000 tonnes in Year-4.
- Initial capital cost, including contingency, is estimated at US$646M, from 1 July 2020.
- Average total cash cost of US$1.16/lb of copper during the first 10 years.
- After-tax NPV, at an 8% discount rate, of US$5.5 billion.
- After-tax internal rate of return (IRR) of 77.0%, and a payback period of 2.3 years.
- Kakula is expected to produce a very-high-grade copper concentrate in excess of 50% copper, with extremely low arsenic levels.

The key results of the Study are summarised in Table 22.2.

Table 22.2 Kakula Results Summary

Item	Unit	Total
Total Processed		
Quantity Milled	kt	109,975
Copper Feed Grade	%	5.22
Total Concentrate Produced		
Copper Concentrate Produced	kt (dry)	8,542
Copper Recovery	%	85.23
Copper Concentrate Grade	%	57.32
Contained Copper in Concentrate	Mlb	10,795
Contained Copper in Concentrate	kt	4,897
Peak Annual Recovered Copper Production	kt	366
Ten Year Average		
Copper Concentrate Produced	kt (dry)	496
Contained Copper in Concentrate	kt	284
Mine-Site Cash Cost	US$/lb Payable Cu	0.52
Total Cash Cost	US$/lb Payable Cu	1.16
Key Financial Results		
Peak Funding	US$M	775
Initial Capital Costs	US$M	646
Expansion Capital Costs	US$M	594
Sustaining Capital Cost	US$M	1,265
Mine Site Cash Cost	US$/lb Payable Cu	0.62
Total Cash Costs After Credits	US$/lb Payable Cu	1.26
Site Operating Costs	US$/t Milled	58.73
After-Tax NPV8%	US$M	5,520
After-Tax IRR	%	77.0
Project Payback Period	Years	2.3
Project Life	Years	21

Table 22.3 summarises the financial results, whilst Table 22.4 summarises mine production, processing, concentrate, and metal production statistics.

Table 22.3 Kakula 2020 FS Financial Results

	Discount Rate (%)	Before Taxation	After Taxation
Net Present Value (US$M)	Undiscounted	16,761	11,595
	4.0	11,258	7,832
	6.0	9,381	6,544
	8.0	7,892	5,520
	10.0	6,698	4,696
	12.0	5,729	4,024
Internal Rate of Return		86.3%	77.0%
Project Payback Period (Years)		2.3	2.3

Table 22.4 Kakula 2020 FS Production and Processing

Item	Unit	Total LOM	Years 1–5	Years 1–10	LOM Average
Total Processed					
Quantity Milled	kt	109,975	4,638	5,345	5,237
Copper Feed Grade	%	5.22	6.56	6.21	5.22
Total Concentrate Produced					
Copper Concentrate Produced	kt (dry)	8,542	454	496	407
Copper Recovery	%	85.23	85.54	85.58	85.23
Copper Concentrate Grade	%	57.32	57.32	57.32	57.32
Contained Copper in Concentrate					
Copper	Mlb	10,795	574	626	514
Copper	kt	4,897	260	284	233
Payable Copper					
Copper	Mlb	10,444	555	606	497
Copper	kt	4,737	252	275	226

The Kakula 2020 FS mill feed and copper grade profile and the concentrate and metal production for the LOM are shown in Figure 22.2 and in Figure 22.3.

Figure 22.2 Kakula 2020 FS Process Production

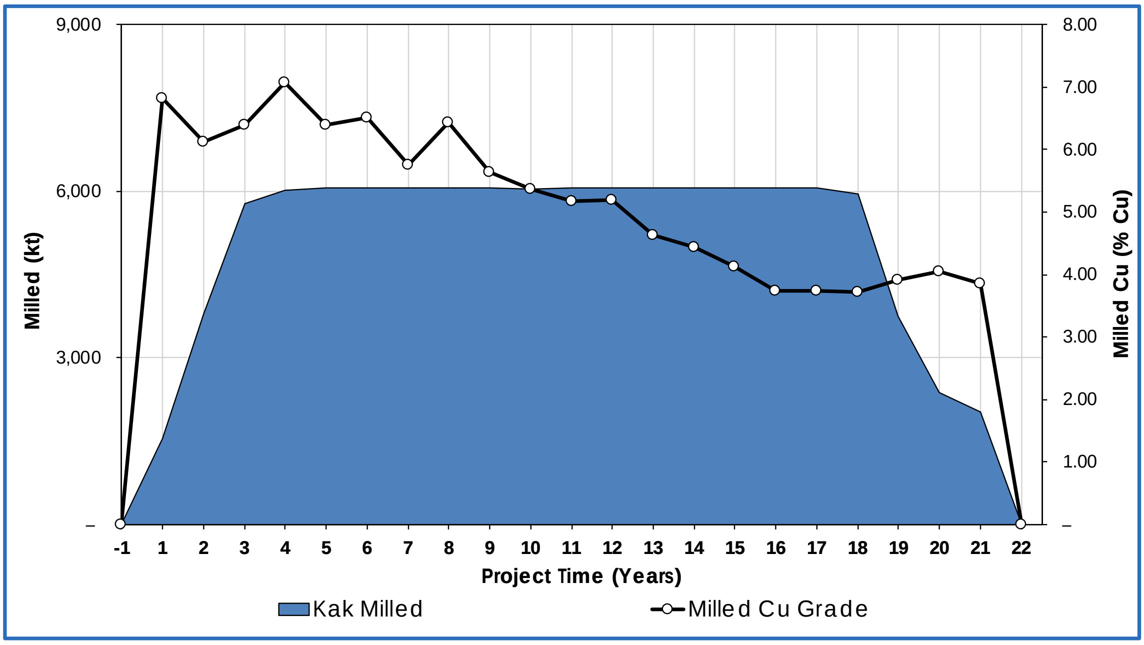


Figure by OreWin, 2020.

Figure 22.3 Kakula 2020 FS Process Production

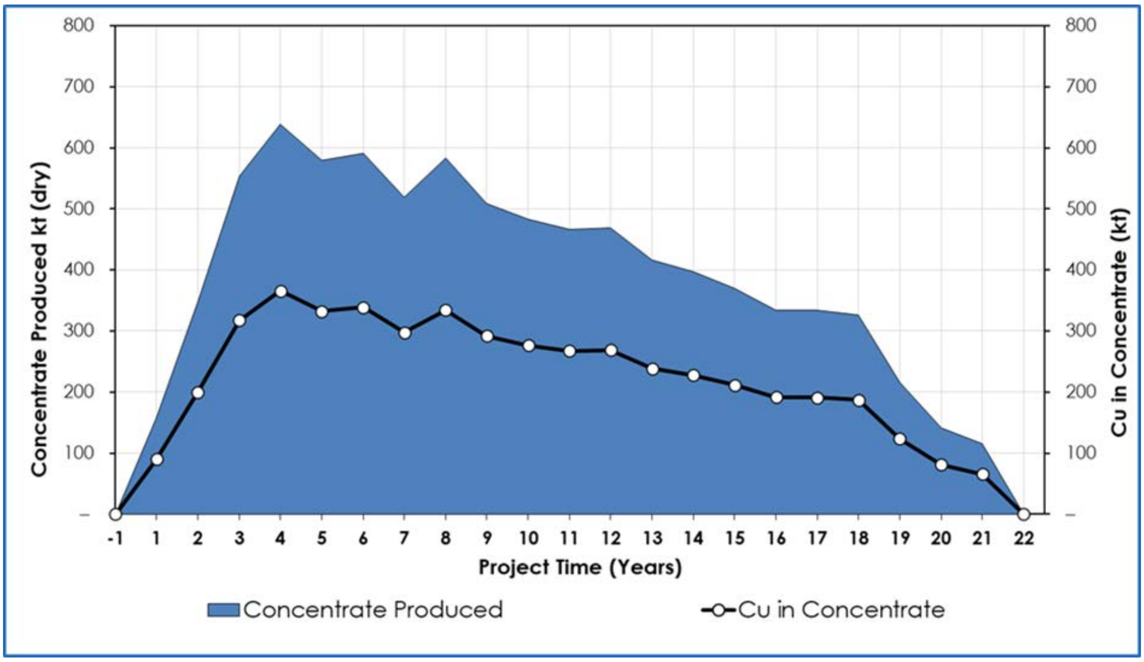


Figure by OreWin, 2020.

22.2.2 Operating Cost and Revenue

The unit operating costs are summarised in Table 22.5. The main components of the revenue and operating costs are summarised in Table 22.6.

Table 22.5 Kakula Unit Operating Costs

	Payable Cu (US$/lb)		
	Years 1–5	Years 1–10	LOM Average
Mine Site	0.48	0.52	0.62
Transport	0.32	0.32	0.32
Treatment and Refining Charges	0.11	0.11	0.11
Royalties and Export Tax	0.20	0.20	0.20
Total Cash Costs	1.12	1.16	1.26

Table 22.6 Kakula 2020 FS Revenue and Operating Costs

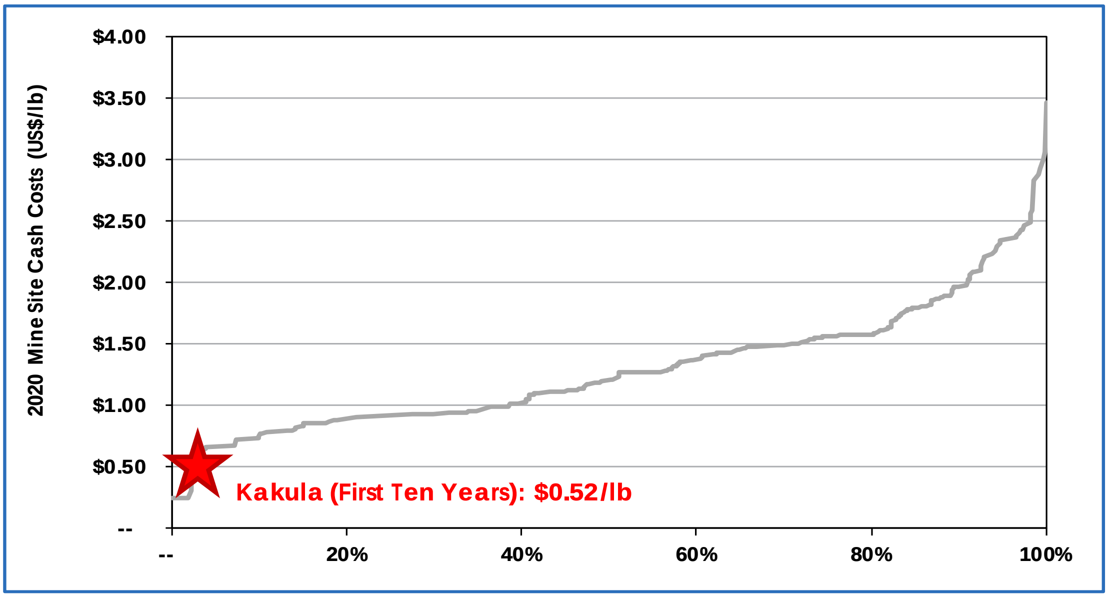
	Total LOM (US$M)	Years 1–5	Years 1–10	LOM Average
		US$/t Milled		
Revenue				
Copper in Concentrate	32,348	369.7	350.9	294.1
Gross Sales Revenue	32,348	369.7	350.9	294.1
Less: Realisation Costs				
Transport	3,383	38.8	36.7	30.8
Treatment and Refining	1,199	13.7	13.0	10.9
Royalties and Export Tax	2,106	24.1	22.9	19.2
Total Realisation Costs	6,689	76.6	72.6	60.8
Net Sales Revenue	25,660	293.1	278.3	233.3
Site Operating Costs				
Underground Mining	4,280	35.4	38.6	38.9
Processing	1,470	14.1	13.4	13.4
General and Administration	758	7.6	7.0	6.9
SNEL Discount	–294	–2.4	–2.6	–2.7
Customs Duties	245	2.1	2.2	2.2
Total	6,459	56.8	58.6	58.7
Net Operating Margin	19,201	236.3	219.7	174.6
Net Operating Margin (%)	74.8%	80.6%	78.9%	74.8%

The average mine-site cash cost during the first 10 years of the Kakula 2020 FS is shown on Wood Mackenzie's industry cost curve in Figure 22.4. 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.

The C1 pro-rata copper cash costs of the Kakula 2020 FS on Wood Mackenzie's industry cost curve is shown in Figure 22.5. C1 pro-rata cash costs that reflect the direct cash costs of producing paid copper incorporating mining, processing, mine-site G&A and offsite realisation costs, having made appropriate allowance for the costs associated with the co-product revenue streams.

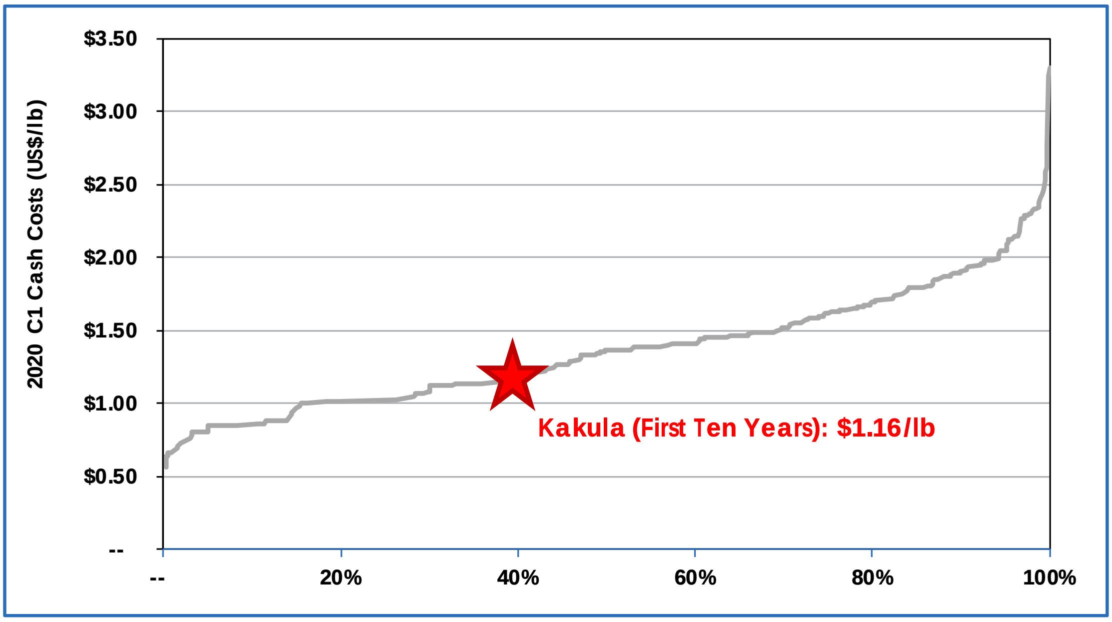
The comparison of cash costs on both charts was not reviewed by Wood Mackenzie prior to filing, and information was sourced from public disclosures.

Figure 22.4 Mine Site Cash Costs (Includes All Operational Costs at Mine Site)



Source: Base data from Wood Mackenzie 2020.

Figure 22.5 C1 Copper Cash Costs



Source: Base data from Wood Mackenzie2020.

22.2.3 Capital Costs

The capital costs for the project are summarised in Table 22.7.

Table 22.7 Kakula 2020 FS Capital Costs

Description	Initial Capital US$M	Expansion Capital US$M	Sustaining Capital US$M	Total US$M
Mining				
Underground Mining	131	202	538	871
Mining Infrastructure and Mobile Equipment	38	16	362	416
Capitalised Preproduction	76	–	–	76
Subtotal	246	218	899	1,363
Off-Site Power				
Power Supply Off Site	36	–	–	36
Subtotal	36	–	–	36
Concentrator and Tailings				
Process Plant	123	128	70	320
Tailings	13	26	88	127
Subtotal	136	154	157	448
Infrastructure				
Surface Infrastructure	69	101	14	184
Subtotal	69	101	14	184
Indirects				
EPCM	35	17	0	53
Owners Cost	66	47	–	114
Customs Duties	8	18	40	66
Closure	–	–	82	82
Subtotal	110	83	122	315
Capital Expenditure Before Contingency	596	556	1,193	2,346
Contingency	50	38	72	159
Capital Expenditure After Contingency	646	594	1,265	2,505

The after-tax net present value (NPV) sensitivity to metal price variation is shown in Table 22.8 for copper prices from US$2.00–US$4.50/lb. Cost sensitivity is shown in Table 22.9.

Table 22.8 Kakula 2020 FS 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**	**4.50**
Undiscounted	4,225	7,519	10,911	11,595	14,353	17,532	19,928
4.0%	2,828	5,072	7,370	7,832	9,704	11,852	13,457
6.0%	2,334	4,227	6,156	6,544	8,117	9,918	11,256
8.0%	1,935	3,551	5,190	5,520	6,857	8,384	9,513
10.0%	1,609	3,005	4,413	4,696	5,845	7,153	8,116
12.0%	1,340	2,558	3,779	4,024	5,022	6,154	6,982
15.0%	1,018	2,028	3,031	3,232	4,052	4,977	5,649
IRR (%)	38.5%	57.9%	74.0%	77.0%	88.9%	100.4%	106.9%

Table 22.9 Kakula 2020 FS Mine Cost Sensitivity

			Change from Base NPV$_{8}$% (US$M)				
Variable	**Units**	**Base Value**	**–25%**	**–10%**	**–**	**10%**	**25%**
Initial Capital Cost	US$M	646	5,612	5,557	5,520	5,483	5,428
Expansion Capital Cost	US$M	594	5,613	5,557	5,520	5,483	5,427
Initial and Expansion Capital Cost	US$M	1,240	5,720	5,594	5,520	5,446	5,335
Site Operating Cost	US$/t Milled	59	6,065	5,732	5,520	5,308	4,990
Treatment and Refining	US$/t and US$/lb Cu	62/0.062	5,622	5,561	5,520	5,479	5,418
Transport	US$/t Conc	364	5,809	5,636	5,520	5,404	5,231

22.2.4 Project Cash Flow

The annual and cumulative cash flows are shown in Figure 22.6 (annual cash flow is shown on the left vertical axis and cumulative cash flow on the right axis).

The revenue, operating cost and capital costs and net cash flow is tabulated in Table 22.10.

Figure 22.6 Kakula 2020 FS Mine Projected Cumulative Cash Flow

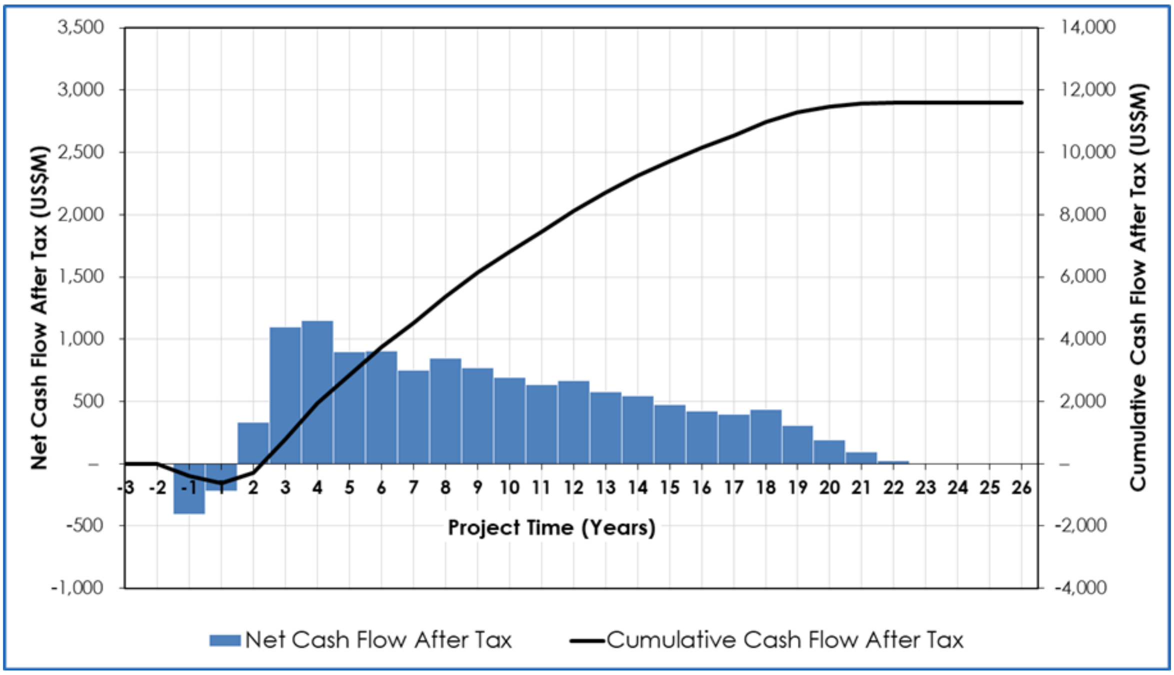


Figure by OreWin, 2020.

OreWin

Table 22.10 Kakula 2020 FS Cash Flow

Cash Flow Statement (US$M)	**Years**										
Cash Flow Measure	**Total**	**–2**	**–1**	**1**	**2**	**3**	**4**	**5**	**6 to 10**	**11 to 20**	**21 to LOM**
Gross Revenue	32,348	–	–	556	1,321	2,078	2,421	2,196	10,183	13,156	436
Realisation Costs	6,689	–	–	120	273	429	500	454	2,104	2,718	90
Net Revenue	25,660	–	–	436	1,048	1,649	1,921	1,743	8,079	10,438	346
Operating Costs											
Mining	4,280	–	–	72	146	181	213	209	1,242	2,123	95
Processing	1,470	–	–	30	61	81	77	77	387	718	38
Tailings	–	–	–	–	–	–	–	–	–	–	–
General and Administration	758	–	–	18	37	40	41	40	200	364	17
Discount on Power	–294	–	–	–5	–9	–13	–14	–15	–81	–150	–7
Customs Duties	245	–	–	4	9	11	12	12	69	121	6
Total Operating Costs	6,459	–	–	120	245	301	329	324	1,817	3,176	148
Operating Surplus / (Deficit)	19,201	–	–	316	803	1,348	1,592	1,419	6,263	7,262	198
Capital Costs											
Initial Capital	646	–	381	264	–	–	–	–	–	–	–
Expansion Capital	594	–	–	205	380	9	–	–	–	–	–
Sustaining Capital	1,265	–	–	–	–	166	150	113	346	407	84
Working Capital	–	–	–	–41	–57	–56	–25	17	27	96	40
VAT	65	–	–22	–29	–24	–11	153	–	–	–	–1
Net Cash Flow Before Tax	16,761	–	–404	–223	342	1,106	1,570	1,322	5,944	6,951	153
Income Tax	5,167	–	–	6	13	21	402	420	1,972	2,295	38
Net Cash Flow After Tax	11,595	–	–404	–229	329	1,085	1,168	902	3,972	4,656	116

22.2.4.1 Project Cash flow Nominal

Figure 22.7 shows the Copper price assumptions in real and nominal terms and projected annual and cumulative cash flow for the Kakula 2020 FS on a nominal basis, using a fixed U.S. annual inflation rate of 2%. Figure 22.8 and Table 22.11 show the Kakula 2020 FS projected annual and cumulative cash flow shown on a nominal basis.

Figure 22.7 Copper Price Assumptions Shown on a Real and Nominal Basis

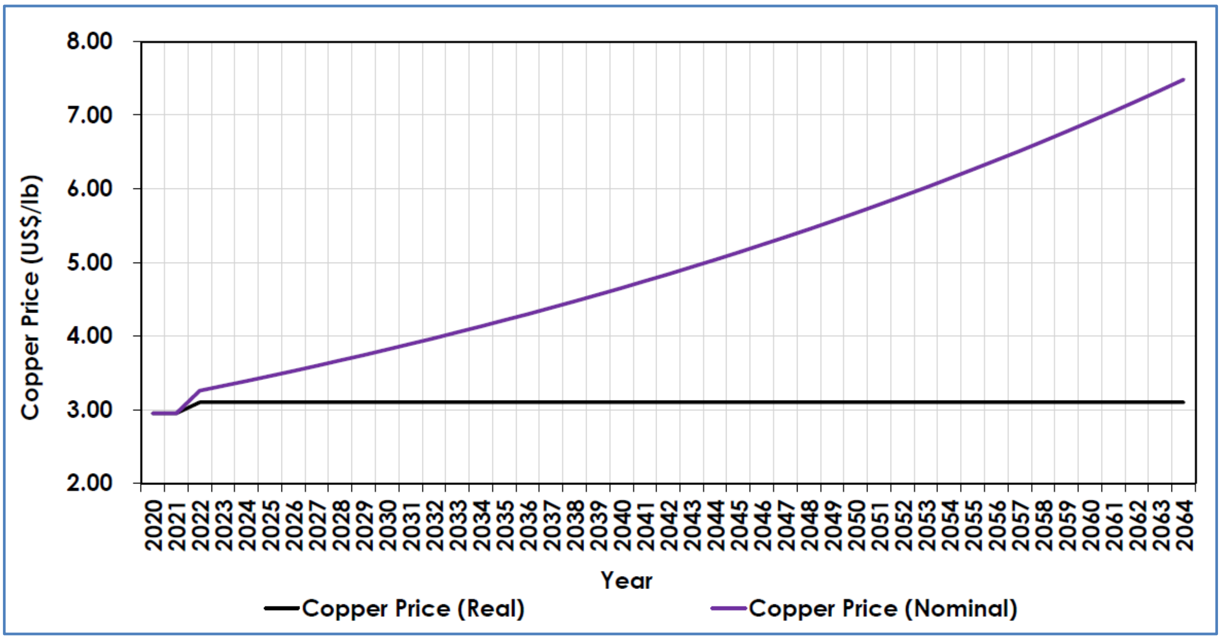


Figure by OreWin, 2020.

Figure 22.8 Kakula 2020 FS Projected Cumulative Cash Flow Nominal basis

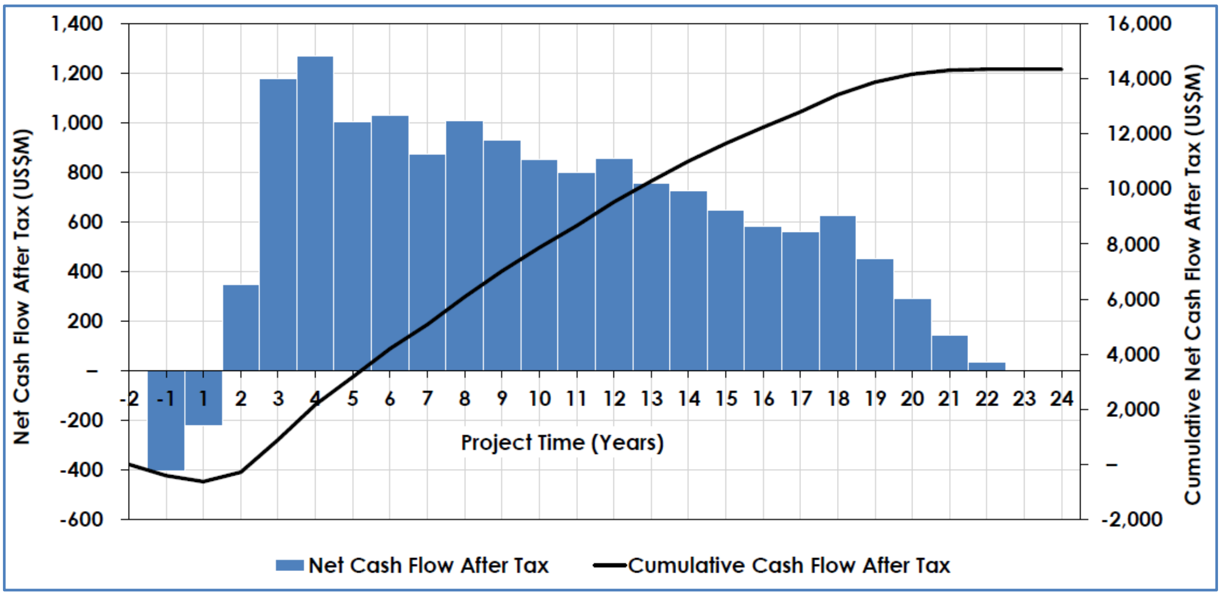


Figure by OreWin, 2020.

Table 22.11 Kakula 2020 FS Cash Flow (Nominal)

Cash Flow Statement (US$M)	Years									
Cash Flow Measure	**Total**	**–2**	**–1**	**1**	**2**	**3**	**4**	**5**	**6 to 10**	**11 to 20**	**21 to LOM**
Gross Revenue	39,781	–	–	556	1,388	2,227	2,647	2,449	12,035	17,812	668
Realisation Costs	8,224	–	–	120	287	460	547	506	2,487	3,680	138
Net Revenue	31,557	–	–	436	1,101	1,767	2,100	1,943	9,548	14,131	530
Operating Costs											
Mining	5,391	–	–	72	154	194	232	233	1,471	2,891	145
Processing	1,849	–	–	30	64	87	84	86	458	981	58
Tailings	–	–	–	–	–	–	–	–	–	–	–
General and Administration	949	–	–	18	39	43	45	45	237	496	27
Discount on Power	–372	–	–	–5	–9	–13	–16	–17	–96	–204	–11
Customs Duties	308	–	–	4	9	12	14	14	81	165	9
Total Operating Costs	8,126	–	–	120	257	322	360	361	2,151	4,328	227
Operating Surplus / (Deficit)	23,431	–	–	316	844	1,445	1,740	1,582	7,397	9,803	303
Capital Costs											
Initial Capital	646	–	381	264	–	–	–	–	–	–	–
Expansion Capital	614	–	–	205	399	10	–	–	–	–	–
Sustaining Capital	1,550	–	–	–	–	178	164	126	408	546	128
Working Capital	60	–	–	–41	–60	–60	–28	19	33	136	62
VAT	77	–	–22	–29	–26	–12	167	–	–	–	–1
Net Cash Flow Before Tax	20,759	–	–404	–223	360	1,185	1,716	1,475	7,022	9,393	235
Income Tax	6,435	–	–	6	14	22	440	469	2,330	3,097	58
Net Cash Flow After Tax	14,324	–	–404	–229	346	1,163	1,276	1,006	4,692	6,296	177

22.3 Kakula-Kansoko 2020 PFS Overview and Results

The Kakula-Kansoko 2020 PFS is a combined schedule comprising of the Kakula 2020 FS (6.0 Mtpa) and a Kansoko 1.6 Mtpa schedule. The mine design for Kansoko is the same as what has been previously called the Kamoa 2019 PFS. Costs were updated using the Kakula 2020 Costs as a basis. 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 2019 PFS.

A probable combined Mineral Reserve of approximately 235.2 Mt grading at 4.47% Cu has been defined in the Kakula and Kansoko mines. A combined 7.6 Mtpa rate reducing to 6.0 Mtpa after the completion of Kakula. The mine design and schedule for Kakula is that of the Kakula 2020 FS. The Kansoko ore zones occur at depths ranging from approximately 60–1,235 m. Access to the mine will be via twin declines. 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.

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$6.6 billion. The Kakula-Kansoko 2020 PFS has an after-tax IRR of 69.0% and a payback period of 2.5 years. The LOM average mine site cash cost is US$0.64/lb of copper.

The estimated initial capital cost, including contingency, is US$695 M, from 1 July 2020. The key results of the Kakula-Kansoko 2020 PFS are summarised in Table 22.12.

Table 22.12 Kakula-Kansoko 2020 PFS Summary

Item	Unit	Total
Total Processed		
Quantity Milled	kt	235,157
Copper Feed Grade	%	4.47
Total Concentrate Produced		
Copper Concentrate Produced	kt (dry)	19,948
Copper Recovery	%	86.27
Copper Concentrate Grade	%	45.49
Contained Copper in Concentrate	Mlb	20,006
Contained Copper in Concentrate	kt	9,075
Peak Annual Contained Metal in Concentrate	kt	427
10-Year Average		
Copper Concentrate Produced	kt (dry)	622
Contained Copper in Concentrate	kt	331
Mine Site Cash Cost	US$/lb	0.55
Total Cash Cost	US$/lb	1.23
Key Financial Results		
Peak Funding	US$M	848
Initial Capital Cost	US$M	695
Expansion Capital Cost	US$M	750
Sustaining Capital Costs	US$M	2,827
LOM Average Mine Site Cash Cost	US$/lb Cu	0.64
LOM Average Total Cash Cost	US$/lb Cu	1.44
Site Operating Cost	US$/t Milled	52.95
After-Tax NPV8%	US$M	6,604
After-Tax IRR	%	69.0
Project Payback Period	Years	2.5
Project Life	Years	37

Table 22.13 Kakula-Kansoko 2020 PFS Financial Results

	Discount Rate (%)	Before Taxation	After Taxation
Net Present Value (US$M)	Undiscounted	27,805	18,373
	4.0	15,562	10,422
	6.0	12,179	8,204
	8.0	9,757	6,604
	10.0	7,967	5,415
	12.0	6,608	4,505
Internal Rate of Return (%)	–	78.5%	69.0%
Project Payback Period (Years)	–	2.5	2.5

Table 22.14 Kakula-Kansoko 2020 PFS Production and Processing

Item	Unit	Total LOM	Years 1–5	Years 1–10	LOM Average
Total Processed					
Quantity Milled	kt	235,157	5,536	6,568	6,356
Copper Feed Grade	%	4.47	6.20	5.87	4.47
Total Concentrate Produced					
Copper Concentrate Produced	kt (dry)	19,948	542	622	539
Copper Concentrate	kt (dry)	19,948	542	622	539
Copper Recovery	%	86.27	85.61	85.84	86.27
Copper Concentrate Grade	%	45.49	54.21	53.27	45.49
Contained Copper in Concentrate					
Copper	Mlb	20,006	648	730	541
Copper	kt	9,075	294	331	245
Payable Copper in Concentrate					
Copper	Mlb	19,356	627	706	523
Copper	kt	8,780	284	320	237
Payable Copper					
Copper	Mlb	19,356	627	706	523
Copper	kt	8,780	284	320	237

Figure 22.9 and Figure 22.10 depict the processing, concentrate and metal production, respectively. Table 22.15 summarises unit operating costs and Table 22.16 provides a breakdown of operating costs and revenue.

Figure 22.9 Kakula-Kansoko 2020 PFS Process Production

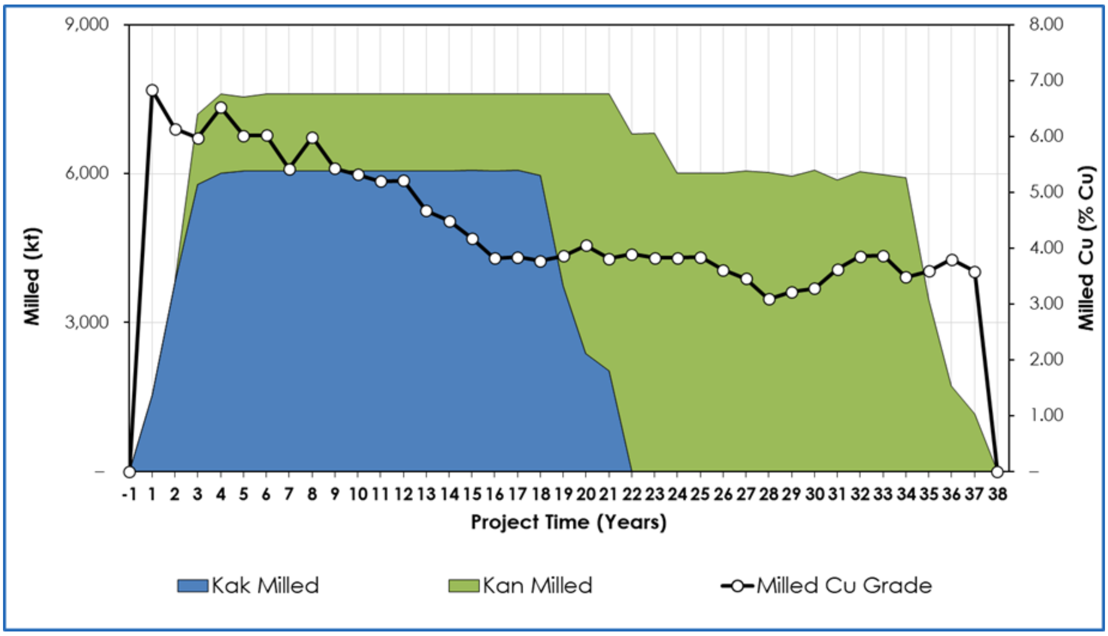


Figure by OreWin, 2020.

Figure 22.10 Kakula-Kansoko 2020 PFS Concentrate and Metal Production

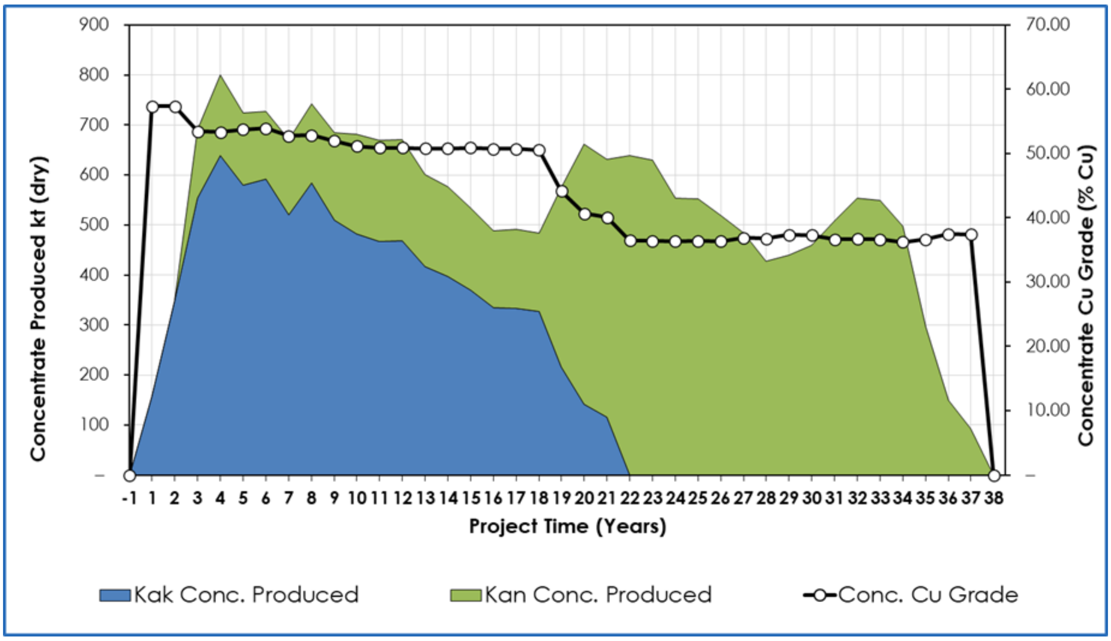


Figure by OreWin, 2020.

Table 22.15 Kakula-Kansoko 2020 PFS Site Operating Costs

	Payable Cu (US$/lb)		
	Years 1–5	**Years 1–10**	**LOM Average**
Mine Site	0.50	0.55	0.64
Transport	0.35	0.35	0.42
Treatment and Refining Charges	0.12	0.12	0.13
Royalties and Export Tax	0.21	0.22	0.25
Total Cash Costs	1.18	1.23	1.44

Table 22.16 Kakula-Kansoko 2020 PFS Revenue and Operating Costs

	Total LOM (US$M)	**Years 1–5**	**Years 1–10**	**LOM Average**
		US$/t Milled		
Revenue				
Copper in Concentrate	59,976	350.2	333.0	255.0
Gross Sales Revenue	59,976	350.2	333.0	255.0
Less: Realisation Costs				
Transport	8,106	39.1	37.8	34.5
Treatment and Refining	2,477	13.3	12.8	10.5
Royalties and Export Tax	4,929	24.1	23.4	21.0
Total Realisation Costs	15,513	76.6	73.9	66.0
Net Sales Revenue	44,463	273.6	259.0	189.1
Site Operating Costs				
Underground Mining	8,134	35.9	38.5	34.6
Processing	3,143	13.7	13.2	13.4
Tailings	45	0.2	0.2	0.2
General and Administration	1,198	7.6	7.2	5.1
SNEL Discount	–545	-2.4	-2.6	-2.3
Customs Duties	476	2.1	2.2	2.0
Total	12,451	57.1	58.6	52.9
Net Operating Margin	32,012	216.5	200.4	136.1
Net Operating Margin (%)	72.0%	79.1%	77.4%	72.0%

The capital costs for the project are detailed in Table 22.17.

Table 22.17 Kakula-Kansoko 2020 PFS Capital Costs

Capital Costs (US$M)	Initial Capital (US$M)	Expansion Capital (US$M)		Sustaining Capital (US$M)	Total (US$M)
		Kakula 6.0 Mtpa / 7.6 Mtpa Plant	Kansoko to 1.6 Mtpa		
Underground Mining					
Underground Mining	158	202	97	1,068	1,525
Capitalised Pre-Production	76	–	–	–	76
Mining Mobile Equipment	55	43	17	922	1,036
Subtotal	289	245	114	1,990	2,638
Off-site Power					
Power Supply Off Site	36	–	–	–	36
Subtotal	36	–	–	–	36
Concentrator and Tailings					
Plant	123	128	–	135	386
Tailings	13	12	–	240	265
Subtotal	136	139	–	375	651
Infrastructure					
Plant Infrastructure	69	101	–	14	184
Conveyor Kansoko to Kakula	–	–	–	95	95
Subtotal	69	101	–	109	279
Indirects					
EPCM	37	15	9	0	62
Owners Cost	67	47	4	–	117
Customs Duties	8	23	–0	89	120
Closure	–	–	–	81	81
Subtotal	113	85	13	170	380
Capital Expenditure Before Contingency	642	571	126	2,644	3,984
Contingency	52	41	12	183	288
Capital Expenditure After Contingency	695	612	139	2,827	4,272

The capital intensity of the Kakula-Kansoko project is compared with other large-scale copper projects in Figure 22.11. 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. Kakula-Kansoko is based on the capital costs incurred in 2019, the capital costs incurred in the six months ended 30 June 2020 and the estimated initial and expansion capital costs from 1 July 2020 in the Kakula-Kansoko 2020 PFS. Kakula-Kansoko's first 10 years' average annual production of copper in concentrate are considered to be its nominal copper production. The Kakula 2020 FS was not reviewed by Wood Mackenzie prior to filing.

Figure 22.11 Capital Intensity for Large Scale Copper Projects

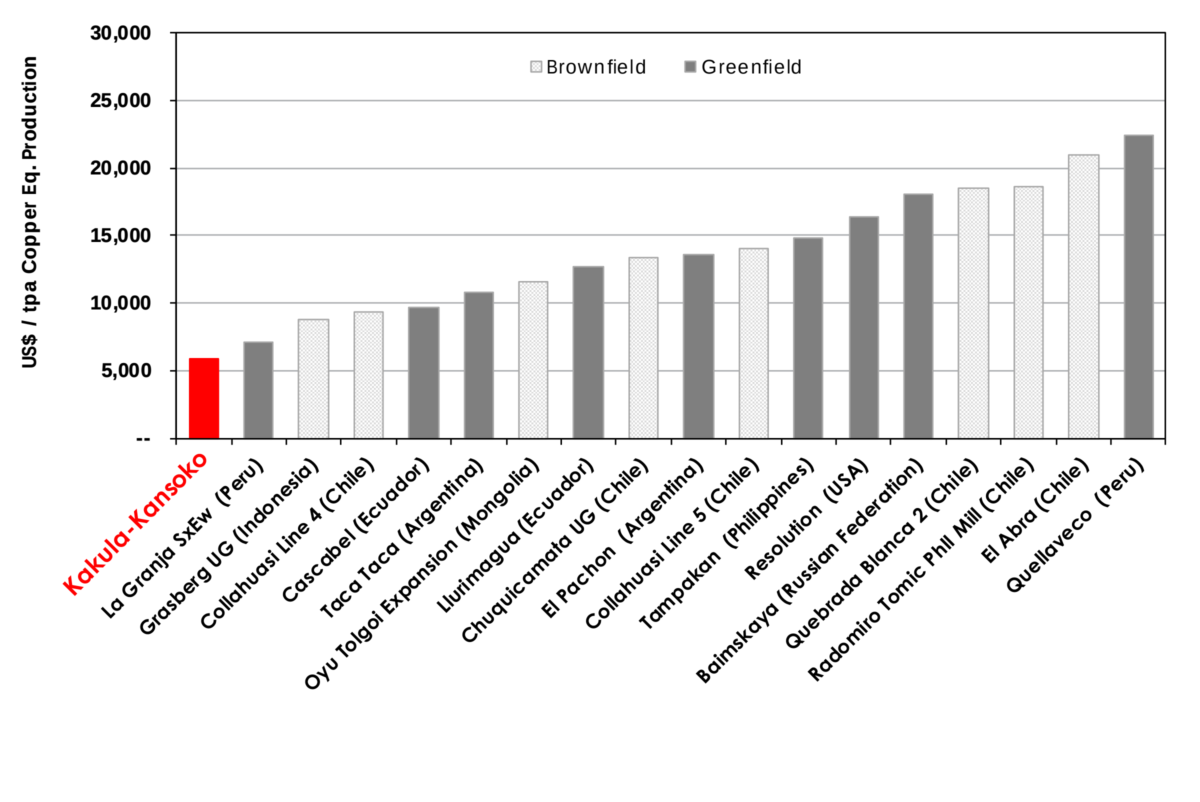


Figure by Ivanhoe, 2020. Source: Wood Mackenzie.

The cash flow sensitivity to metal price variation is shown in Table 22.18, for copper prices from US$2.00/lb Cu to US$4.50/lb.

The sensitivity of After Tax NPV8% to initial capital cost, expansion capital cost, direct operating costs, treatment and refining, and transport are shown in Table 22.19. The table shows the change in the base case After Tax NPV8% of US$6,604M. The sensitivity to treatment and refining applies the concentrate treatment charges of US$62/t concentrate and concentrate refining charge of US$0.062/lb Cu. The sensitivity to transport applies the costs via road (US$364/t) and via rail (US$353/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 22.18 Kakula-Kansoko 2020 PFS 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**	**4.50**
Undiscounted	4,758	10,753	17,101	18,373	23,487	29,393	33,873
4.0%	2,847	6,181	9,714	10,422	13,275	16,560	19,031
6.0%	2,241	4,871	7,648	8,204	10,448	13,026	14,957
8.0%	1,774	3,911	6,156	6,604	8,419	10,498	12,047
10.0%	1,409	3,188	5,044	5,415	6,915	8,631	9,902
12.0%	1,117	2,629	4,194	4,505	5,770	7,212	8,274
15.0%	781	2,000	3,247	3,495	4,501	5,644	6,480
IRR (%)	29.5%	49.2%	66.0%	69.0%	81.0%	93.0%	99.8%

Table 22.19 Kakula-Kansoko 2020 PFS Additional Sensitivities

			Change from Base NPV_8% (US$M)				
Variable	**Units**	**Base Value**	**–25%**	**–10%**	**–**	**10%**	**25%**
Initial Capital Cost	US$M	695	6,703	6,644	6,604	6,565	6,505
Expansion Capital Cost	US$M	750	6,720	6,650	6,604	6,558	6,489
Initial and Expansion Capital Cost	US$M	1,445	6,819	6,690	6,604	6,519	6,392
Site Operating Cost	US$/t Milled	53	7,350	6,906	6,604	6,293	5,828
Treatment and Refining	US$/t and US$/lb Cu	62/0.062	6,751	6,663	6,604	6,546	6,458
Transport	US$/t Conc	364	7,061	6,787	6,604	6,422	6,148

The annual and cumulative cash flows are shown in Figure 22.12 (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 22.20.

Figure 22.12 Kakula-Kansoko 2020 PFS Projected Cumulative Cash Flow

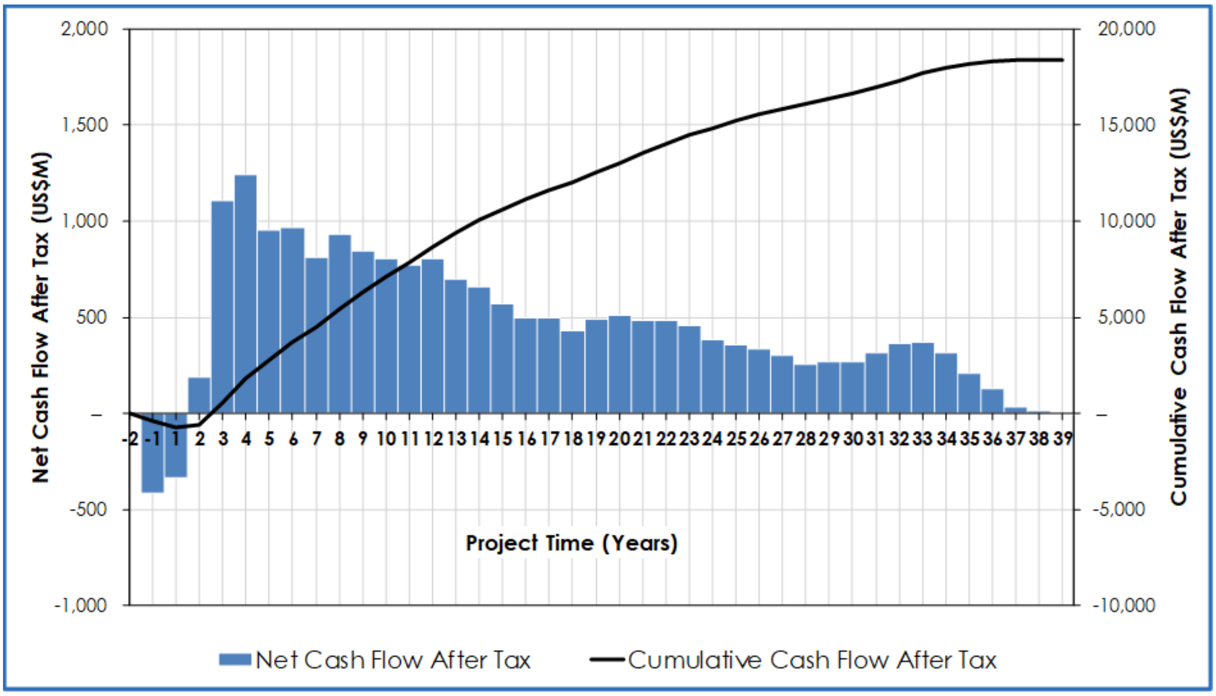


Figure by OreWin, 2020.

Table 22.20 Kakula-Kansoko 2020 PFS Cash Flow

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	59,976	–	–	556	1,321	2,420	2,824	2,572	12,176	18,619	19,487
Realisation Costs	15,513	–	–	120	273	536	626	566	2,737	4,481	6,175
Net Revenue	44,463	–	–	436	1,048	1,885	2,198	2,006	9,439	14,138	13,313
Operating Costs											
Mining	8,134	–	–	79	172	221	265	258	1,532	2,828	2,780
Processing	3,143	–	–	30	61	94	97	97	487	974	1,303
Tailings	45	–	–	1	1	1	1	1	6	13	21
General and Administration	1,198	–	–	18	37	52	54	50	261	491	235
Discount on Power	-545	–	–	–5	–9	–16	–18	–19	–102	–206	–169
Customs Duties	476	–	–	5	10	14	16	15	86	163	167
Total Operating Costs	12,451	–	–	128	272	366	414	402	2,270	4,263	4,337
Operating Surplus / (Deficit)	32,012	–	–	309	776	1,519	1,784	1,604	7,169	9,875	8,976
Capital Costs											
Initial Capital	695	–	390	305	–	–	–	–	–	–	–
Expansion Capital	750	–	0	253	488	10	–	–	–	–	–
Sustaining Capital	2,827	–	–	–	–	280	298	218	594	781	656
Working Capital	–	–	–	-41	-57	-81	-30	19	20	39	131
VAT	65	–	–23	-35	-31	-19	174	–	–	–	-1
Net Cash Flow Before Tax	27,805	–	–413	-325	201	1,129	1,630	1,404	6,596	9,134	8,450
Income Tax	9,432	–	–	6	13	24	388	454	2,238	3,196	3,114
Net Cash Flow After Tax	18,373	–	–413	-330	188	1,105	1,243	950	4,357	5,938	5,336

Figure 22.13 shows the copper price assumptions in real and nominal terms and projected annual and cumulative cash flow for the Kakula-Kansoko 2020 PFS shown on a nominal basis, using a fixed U.S. annual inflation rate of 2%. Figure 22.14 and Table 22.11 show the Kakula-Kansoko 2020 PFS projected annual and cumulative cash flow shown on a nominal basis.

Figure 22.13 Copper Price Assumptions Shown on a Real and Nominal Basis

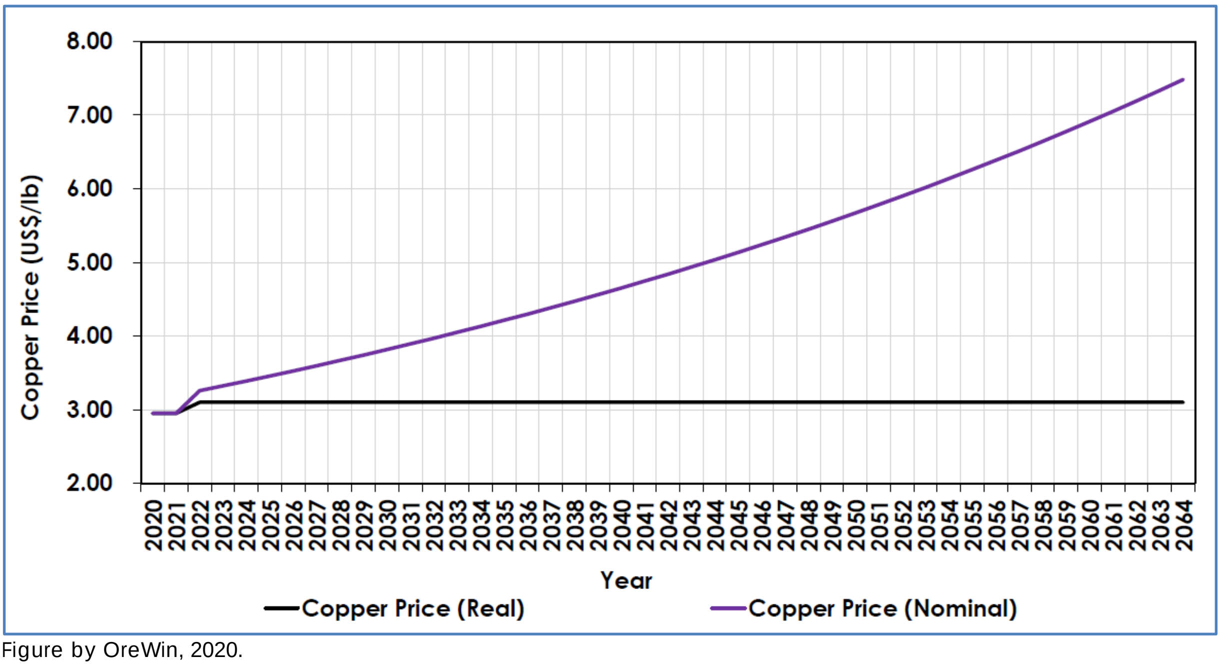


Figure by OreWin, 2020.

Figure 22.14 Kakula-Kansoko 2020 PFS Projected Cumulative Cash Flow Nominal Basis

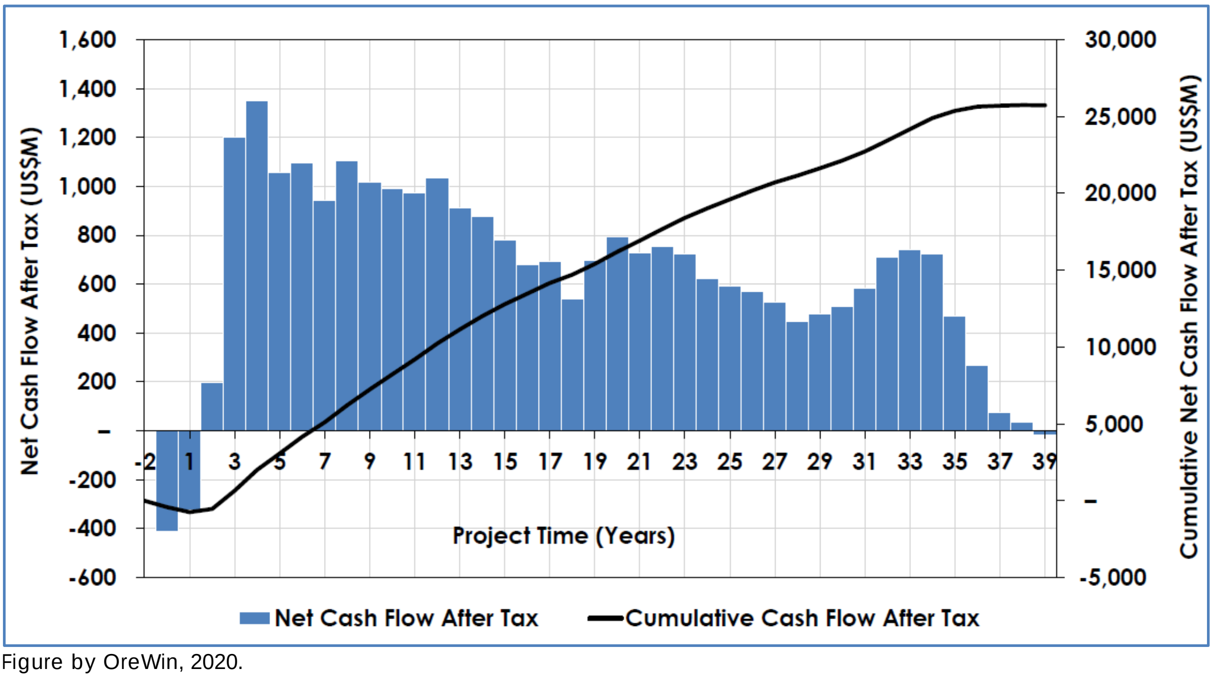


Figure by OreWin, 2020.

Table 22.21 Kakula-Kansoko 2020 PFS Cash Flow (Nominal)

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	84,566	–	–	556	1,388	2,594	3,087	2,868	14,401	25,450	34,223
Realisation Costs	22,530	–	–	120	287	574	684	631	3,239	6,145	10,851
Net Revenue	62,036	–	–	436	1,101	2,020	2,403	2,237	11,162	19,305	23,372
Operating Costs											
Mining	11,652	–	–	79	180	237	289	288	1,814	3,877	4,889
Processing	4,625	–	–	30	64	101	106	108	577	1,339	2,299
Tailings	68	–	–	1	1	1	1	1	7	17	38
General and Administration	1,626	–	–	18	39	56	59	55	309	672	418
Discount on Power	-758	–	–	-5	-10	-17	-20	-21	-121	-284	-279
Customs Duties	684	–	–	5	10	15	17	17	102	224	294
Total Operating Costs	17,898	–	–	128	285	392	453	449	2,687	5,846	7,658
Operating Surplus / (Deficit)	44,139	–	–	309	816	1,628	1,951	1,788	8,475	13,459	15,714
Capital Costs											
Initial Capital	695	–	390	305	–	–	–	–	–	–	–
Expansion Capital	776	–	0	253	512	10	–	–	–	–	–
Sustaining Capital	3,822	–	–	–	–	301	325	243	700	1,075	1,177
Working Capital	126	–	–	-41	-60	-87	-33	21	24	51	251
VAT	78	–	-23	-35	-33	-20	190	–	–	–	-1
Net Cash Flow Before Tax	39,050	–	-413	-325	211	1,210	1,782	1,566	7,798	12,435	14,786
Income Tax	13,423	–	–	6	14	26	424	506	2,647	4,358	5,443
Net Cash Flow After Tax	25,627	–	-413	-330	197	1,184	1,359	1,060	5,152	8,077	9,343

23 ADJACENT PROPERTIES

There are no adjacent properties relevant to this Report.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Kamoa-Kakula 2020 PEA

The Kamoa-Kakula 2020 PEA analyses a production case with an expansion of the Kakula concentrator processing facilities, and associated infrastructure to 19 Mtpa and includes a smelter and eight separate underground mining operations with associated capital and operating costs. Overview of deposits included within the Kakula 2020 FS (outlined by blue dotted line), Kakula-Kansoko 2020 PFS (outlined by purple dotted line) and Kamoa-Kakula 2020 PEA (outlined by green dotted line) is shown in Figure 24.1. The eight mines ranked by their relative values are:

- Kakula Mine (FS 6.0 Mtpa).
- Kansoko Mine (PFS 1.6 Mtpa to 6.0 Mtpa).
- Kakula West Mine (PEA 6.0 Mtpa).
- Kamoa North Mine 1 (PEA 6.0 Mtpa).
- Kamoa North Mine 2 (PEA 6.0 Mtpa).
- Kamoa North Mine 3 (PEA 6.0 Mtpa).
- Kamoa North Mine 4 (PEA 3.0 Mtpa).
- Kamoa North Mine 5 (PEA 1.0 Mtpa).

Figure 24.1 Kamoa-Kakula 2020 PEA Mining Locations

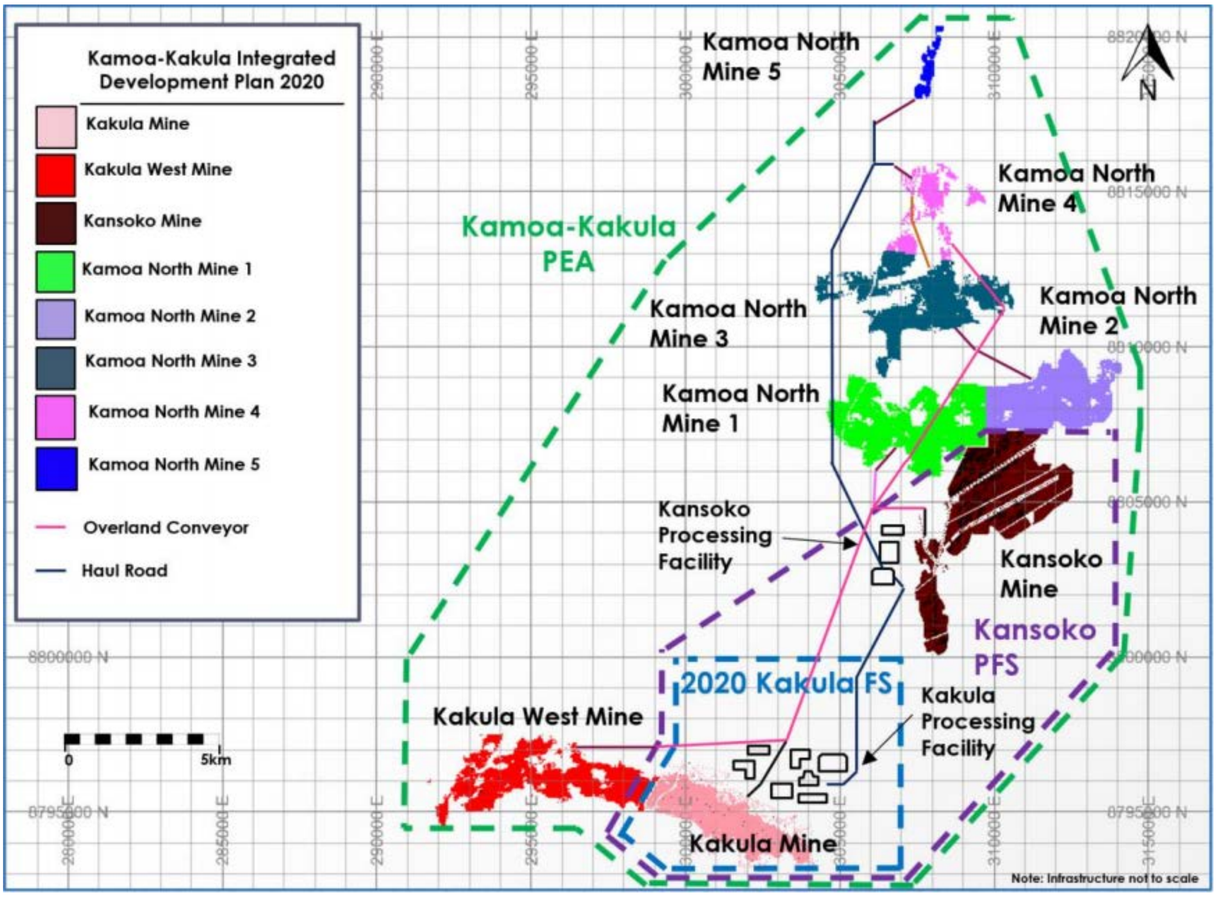


Figure by OreWin, 2020.

The Kamoa-Kakula 2020 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 2020 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 North Mine 1, Kamoa North Mine 2, Kamoa North Mine 3, Kamoa North Mine 4 and Kamoa North Mine 5 for the PEA analyses and have been prepared using the Mineral Resources stated in the Kamoa-Kakula 2020 Resource Update.

The potential development scenarios at the Kamoa-Kakula Project include the Kamoa-Kakula IDP20 development scenario shown in Figure 24.2.

The Kamoa-Kakula 2020 PEA assesses an alternative development option of mining several deposits on the Kamoa-Kakula Project as an integrated, 19 Mtpa mining, processing and smelting complex, built in three stages. This scenario envisages the construction and operation of eight separate mines: first, an initial 6.0 Mtpa mining operation would be established at the Kakula Mine on the Kakula Deposit; this is followed by a subsequent, separate 6.0 Mtpa mining operation at the Kansoko Mine using the existing twin declines that were completed in 2017; a third 6.0 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 five other mines in the Kamoa North area to maintain throughput of 19 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 and Kansoko. 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.

Figure 24.2 Kamoa-Kakula 2020 PEA Long-Term Development Scenario

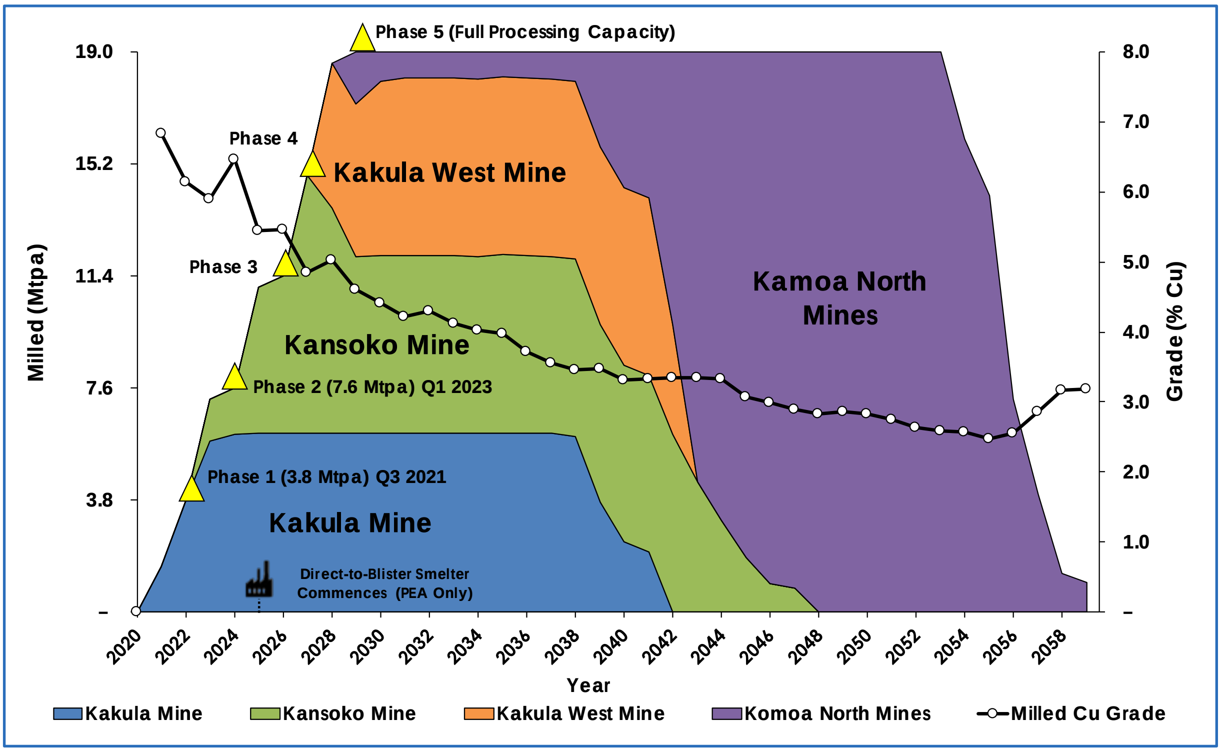


Figure by OreWin, 2020.

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 2020 PEA 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 2020 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 2020 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. 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 IDP20 Site Plan

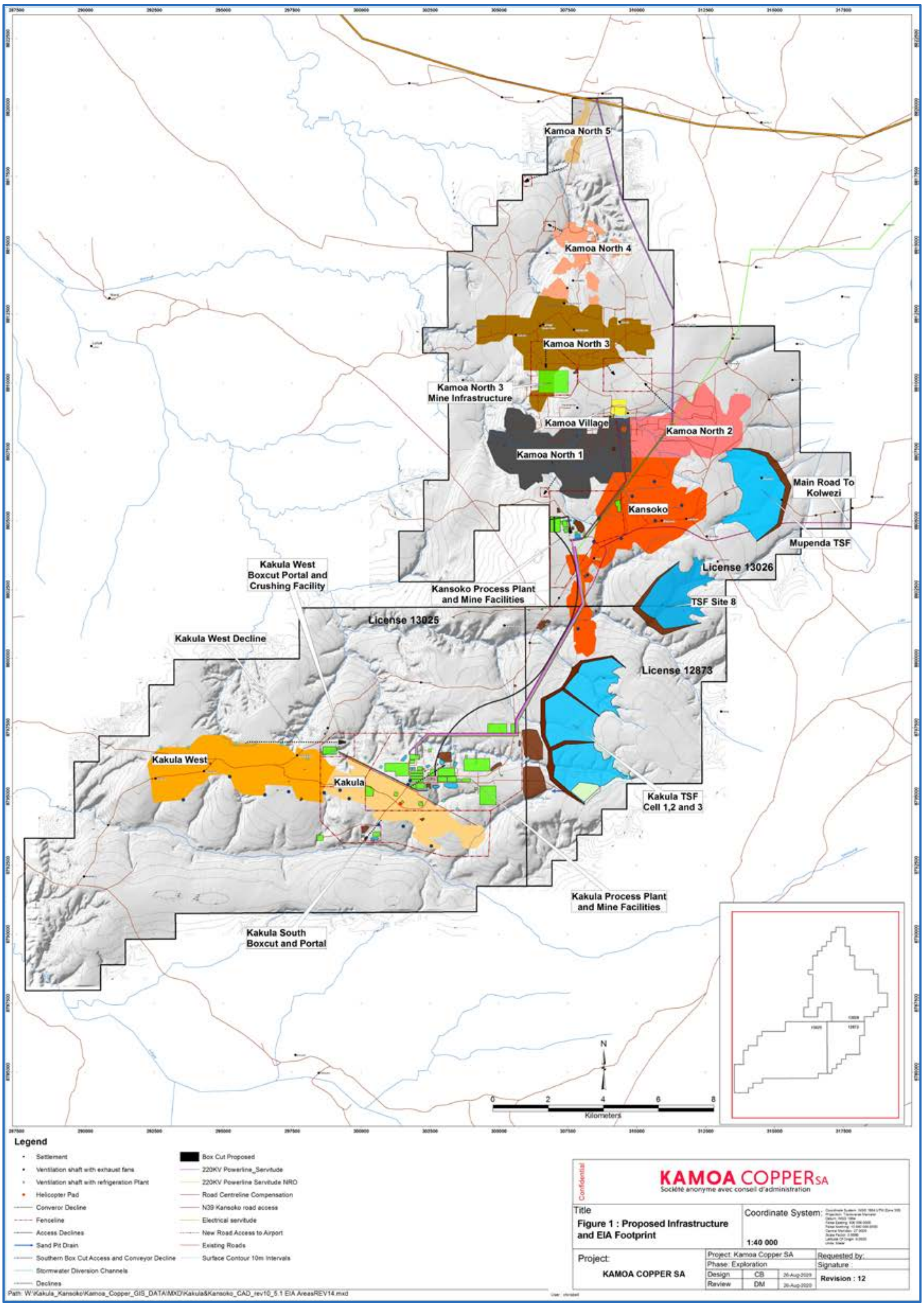


Figure by Kamoa Copper SA, 2020.

24.2 Kamoa-Kakula 2020 PEA Assumptions

24.2.1 Economic Assumptions

24.2.1.1 Pricing and Discount Rate Assumptions

The Project level valuation model begins on 1 July 2020. It is presented in Q2'20 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 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 96.75%.

The copper concentrate assumes an $62/t treatment charge and refining charge of US$0.062/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$364/t via road to Ndola and rail to Durban for shipping for the first two years of production and thereafter US$353/t via rail through Lobito.

24.2.1.2 Taxation

In the DRC, companies that are holders of mining rights are subject to 30% taxation on net income. The economic model applies this taxation rate after accounting for operating costs and depreciation on capital investments.

Provincial taxation on copper concentrate and national export tax is applied in the economic model on copper concentrate production. These taxes are applied independently of capital and operating costs.

24.2.1.3 Royalties

The royalty is due upon the sale of the product and is calculated at 3.5% of the gross commercial value of non-ferrous metals.

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

24.2.1.4 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 corporate income tax (CIT) based on tax at 30% on net income, excluding transportation costs.

A minimum tax of 1% of revenue, which is deductible from the CIT basis, is payable in the event that a mining company is in a loss position or offsetting previous losses carried forwards (see below), and withholding tax on distributions are subject to 10% tax at the shareholder's level. In addition, as from 1 January 2014, the minimum amount of tax payable by mining companies in a year is 1% of the calculated revenue for that specific year.

Tax Holidays

The DRC tax legislation does not currently provide for any tax holiday incentives.

Tax Losses

Tax losses from a financial year may be deducted from profits earned in subsequent years up to the fifth year following the loss-making period. The aggregate exploration expenditure may be claimed.

Research and Development Costs

Research and development costs capitalised during the exploration and construction phases may be amortised over a period of two years from first production, with losses resulting from such an amortisation allowed to be carried forwards. These costs include exploration, owners' costs, certain underground development costs, and interest paid on shareholders loans.

Depreciation

Specific mining assets dedicated to mining operations, with useful lives between 4–20 years are depreciated on a straight line basis.

Non-mining assets are depreciated in accordance with the common law. The common low provides different depreciation rates for various assets, e.g. 10-years for plant and equipment.

Value Added Tax

Value added tax (VAT) came into effect in the DRC in January 2012. VAT is levied on all supplies of goods and services at a rate of 16% and is not levied on any capital asset movements.

VAT Exoneration

Holders of mining rights are currently entitled to exoneration for certain import duties and import VAT for materials and equipment imported for construction of a mine and related infrastructure in accordance with the VAT Act, subject to any unforeseen changes in the law. Kamoa Copper SA has successfully received the exoneration in the past in terms of the mining code and it expects to receive such exoneration for most imports for project construction in future in compliance with the mining code.

Customs / Import Duties

Imports of equipment are subject to an entry fee at the rate of:

- 2% before the beginning of the mining operations.
- 5% until the end of the third year of the first production.

All intermediate goods and other consumables are taxed at the rate of 10% of tariffs, except fuels and lubricants for mining activities which are subject to the rate of 5%. The common law system is applicable to all imports after five years from the date of the granting of the mining licence.

Export Taxes

Fees and royalties for services rendered – claimed by DRC Agencies and specific public services – is limited to 1.0% of the gross commercial value of the export. In addition, a levy of 0.25% of the gross commercial value is charged for Redevance Informatique.

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 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 this 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.

24.2.1.5 Sunk Costs

The estimate excludes all sunk costs up to 30 June 2020.

24.3 Kamoa-Kakula 2020 PEA Results

The Kamoa-Kakula 2020 PEA assesses an alternative development option of mining several deposits on the Kamoa-Kakula Project as an integrated, 19 Mtpa mining, processing and smelting complex, built in three stages. This scenario envisages the construction and operation of three separate mines: first, an initial 6.0 Mtpa mining operation would be established at the Kakula Mine on the Kakula Deposit; this is followed by a subsequent, separate 6.0 Mtpa mining operation at the Kansoko Mine using the existing twin declines that were completed in 2017; a third 6.0 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 five other mines in the Kamoa North area to maintain throughput of 19 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 and Kansoko. Included in this scenario is the construction of a direct-to-blister flash copper smelter with a capacity of 1 Mtpa of copper concentrate. The development scenario of the Kamoa-Kakula 2020 PEA is shown in Figure 24.4.

Figure 24.4 Kamoa-Kakula 2020 PEA Development Scenario

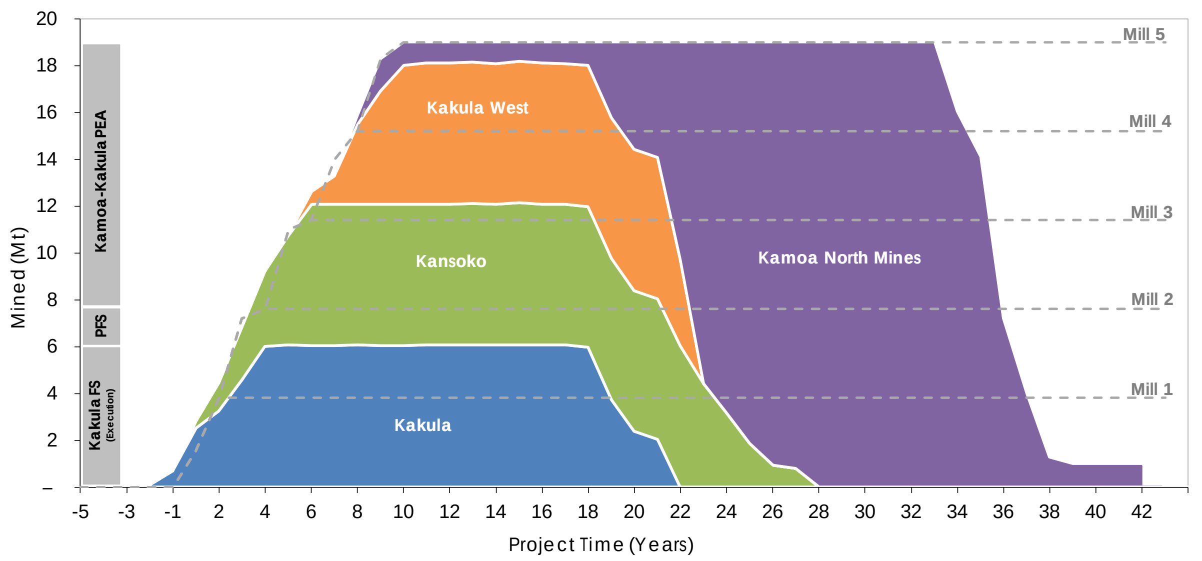


Figure by OreWin, 2020.

A summary of the key results for the Kamoa-Kakula 2020 PEA scenario are:

- Very-high-grade initial phase of production is projected to have a grade of 6.8% copper in the first year of production and an average grade of 5.13% copper over the initial 10-years of operations, resulting in estimated average annual copper production of 501,000 tonnes.
- Initial capital cost, including contingency, is estimated at US$715M.
- Average total cash cost of US$1.07/lb of copper during the first 10-years, including sulfuric acid credits.
- After-tax NPV, at an 8% discount rate, of US$11.12b.
- After-tax internal rate of return (IRR) of 56.2%, and a payback period of 3.6-years.

The LOM production scenario provides for 597.6 Mt to be mined at an average grade of 3.63% copper, producing 43 Mt of high-grade copper concentrate, containing approximately 40.9 b 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$11.1 b. It has an after-tax IRR of 56.2% and a payback period of 3.6-years.

The estimated initial capital cost, including contingency, is US$715M. The capital expenditure for off-site power, which is included in the initial capital cost, includes a US$36M 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 2020 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 2020 PEA are summarised in Table 24.2. The mining production statistics are shown in Table 24.3. The Kamoa-Kakula 2020 PEA 19 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 2020 PEA as part of the Kamoa-Kakula IDP20 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 IDP20 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 IDP20. 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 2020 PEA Financial Results

	Discount Rate (%)	Before Taxation	After Taxation
Net Present Value (US$M)	Undiscounted	56,564	37,844
	4.0	29,097	19,416
	6.0	21,816	14,520
	8.0	16,756	11,117
	10.0	13,136	8,681
	12.0	10,476	6,891
Internal Rate of Return (%)	–	66.4%	56.2%
Project Payback Period (Years)	–	3.4	3.6

Table 24.2 Kamoa-Kakula 2020 PEA Results Summary for 19 Mtpa Production

Item	Unit	Total
Total Processed		
Quantity Milled	kt	597,621
Copper Feed Grade	%	3.63
Total Concentrate Produced		
Copper Concentrate Produced	kt (dry)	42,818
Copper Recovery	%	86.42
Copper Concentrate Grade	%	43.76
Contained Copper in Concentrate - External Smelter	Mlb	13,251
Contained Copper in Concentrate - External Smelter	kt	6,010
Contained Copper in Blister - Internal Smelter	Mlb	27,641
Contained Copper in Blister - Internal Smelter	kt	12,538
Peak Annual Recovered Copper Production	kt	805
10-Year Average		
Copper Concentrate Produced	kt (dry)	1,043
Contained Copper in Conc. - External Smelter	kt	248
Contained Copper in Blister - Internal Smelter	kt	253
Mine-Site Cash Cost	US$/lb Cu	0.65
Total Cash Cost	US$/lb Cu	1.07
Key Financial Results		
Peak Funding	US$M	784
Initial Capital Cost	US$M	715
Expansion Capital Cost	US$M	4,461
Sustaining Capital Cost	US$M	11,958
LOM Average Mine Site Cash Cost	US$/lb Cu	0.92
LOM Average Total Cash Cost	US$/lb Cu	1.28
Site Operating Cost	US$/t Milled	62.44
After-Tax NPV8%	US$M	11,117
After-Tax IRR	%	56.2
Project Payback Period	Years	3.6
Project Life	Years	43

Table 24.3 Kamoa-Kakula 2020 PEA Production and Processing

Item	Unit	Total LOM	Years 1–5	Years 1–10	LOM Average
Total Processed					
Quantity Milled	kt	597,621	6,227	11,394	13,898
Copper Feed Grade	%	3.63	5.95	5.13	3.63
Total Concentrate Produced					
Copper Concentrate Produced	kt (dry)	42,818	613	1,043	996
Copper Concentrate - External Smelter	kt (dry)	11,944	413	443	278
Copper Concentrate - Internal Smelter	kt (dry)	30,874	200	600	718
Copper Recovery	%	86.42	85.98	86.44	86.42
Copper Concentrate Grade	%	43.76	52.02	48.38	43.76
Contained Copper in Concentrate - External Smelter					
Copper	Mlb	13,251	496	548	308
Copper	kt	6,010	225	248	140
Payable Copper in Concentrate - External Smelter					
Copper	Mlb	12,820	480	530	298
Copper	kt	5,815	218	240	135
Contained Copper in Blister - Internal Smelter					
Copper	Mlb	27,641	204	557	643
Copper	kt	12,538	92	253	292
Payable Copper in Blister - Internal Smelter					
Copper	Mlb	27,558	203	555	641
Copper	kt	12,500	92	252	291
Payable Copper					
Copper	Mlb	40,378	683	1,085	939
Copper	kt	18,315	310	492	426

Figure 24.5 Kamoa-Kakula 2020 PEA Process Production

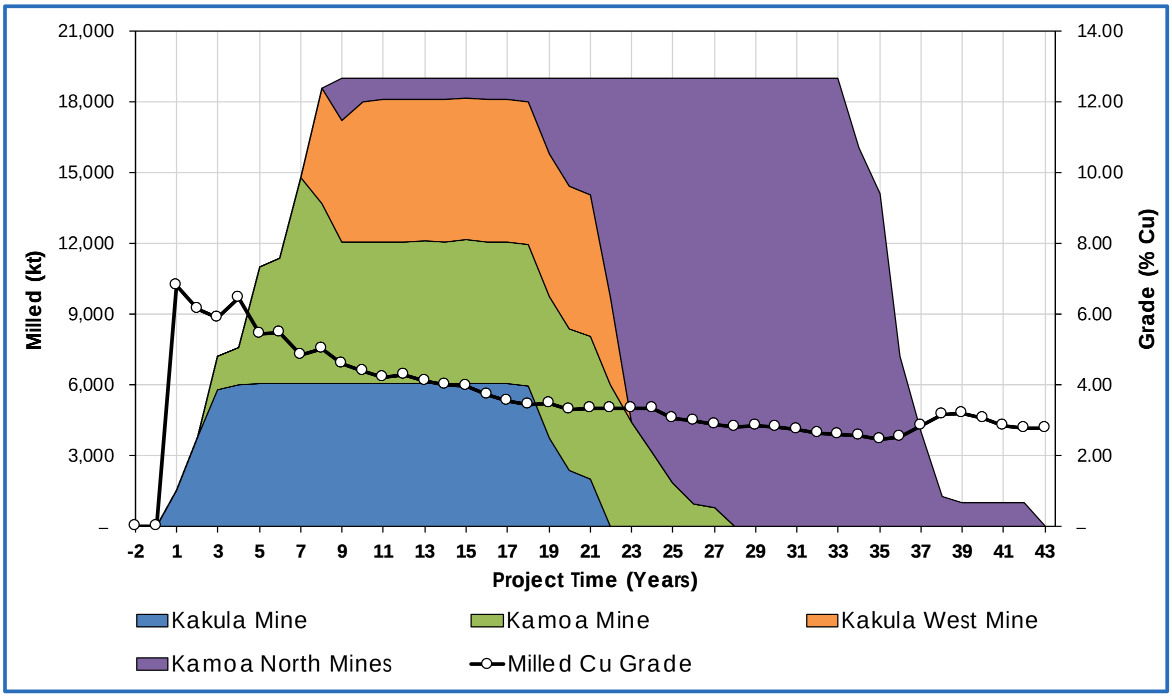


Figure by OreWin, 2020.

Figure 24.6 Kamoa-Kakula 2020 PEA Concentrate and Metal Production

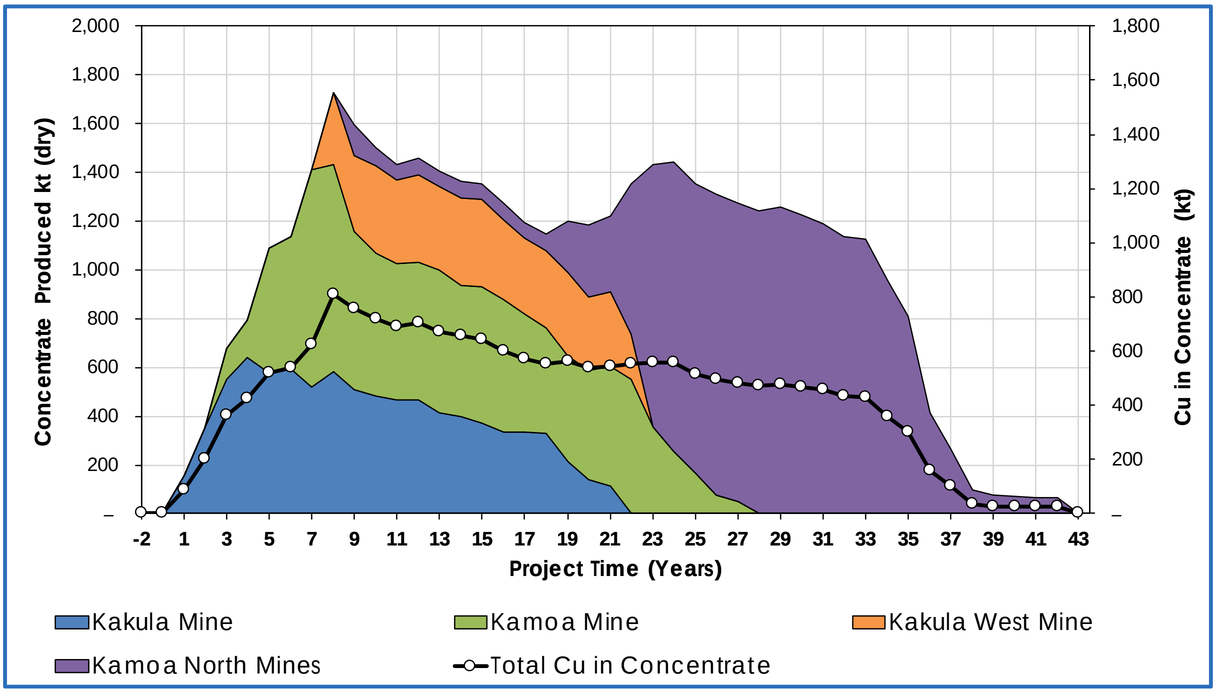


Figure by OreWin, 2020.

The Kamoa-Kakula 2020 PEA as part of the Kamoa-Kakula IDP20 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 IDP20 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 IDP20. 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 highest producers by paid copper production. The Kamoa-Kakula 2020 PEA production is from the projected peak copper production which occurs in Year-8. Figure 24.8 shows the top 10 largest new greenfield copper projects defined as the 10 largest greenfield copper projects classified by Wood Mackenzie as "base case" or "probable" and ranked by nominal copper production (with the Kamoa-Kakula 2020 PEA and Kakula 2020 FS's respective first 10 years' average annual production of copper in concentrate considered to be its nominal copper production). The Kamoa-Kakula IDP20 was not reviewed by Wood Mackenzie prior to filing.

Table 24.4 Kamoa-Kakula 2020 PEA Unit Operating Costs

	Payable Copper (US$/lb)		
	Years 1–5	**Years 1–10**	**LOM Average**
Mine Site	0.56	0.57	0.81
Smelter	0.04	0.08	0.11
Transport	0.29	0.24	0.23
Treatment and Refining Charges	0.10	0.08	0.07
Royalties and Export Tax	0.19	0.18	0.17
Total Cash Costs	1.18	1.15	1.40
Sulfuric Acid Credits	0.04	0.09	0.12
Total Cash Costs After Credits	1.14	1.07	1.28

Figure 24.7 2025 Predicted World Copper Producer Production

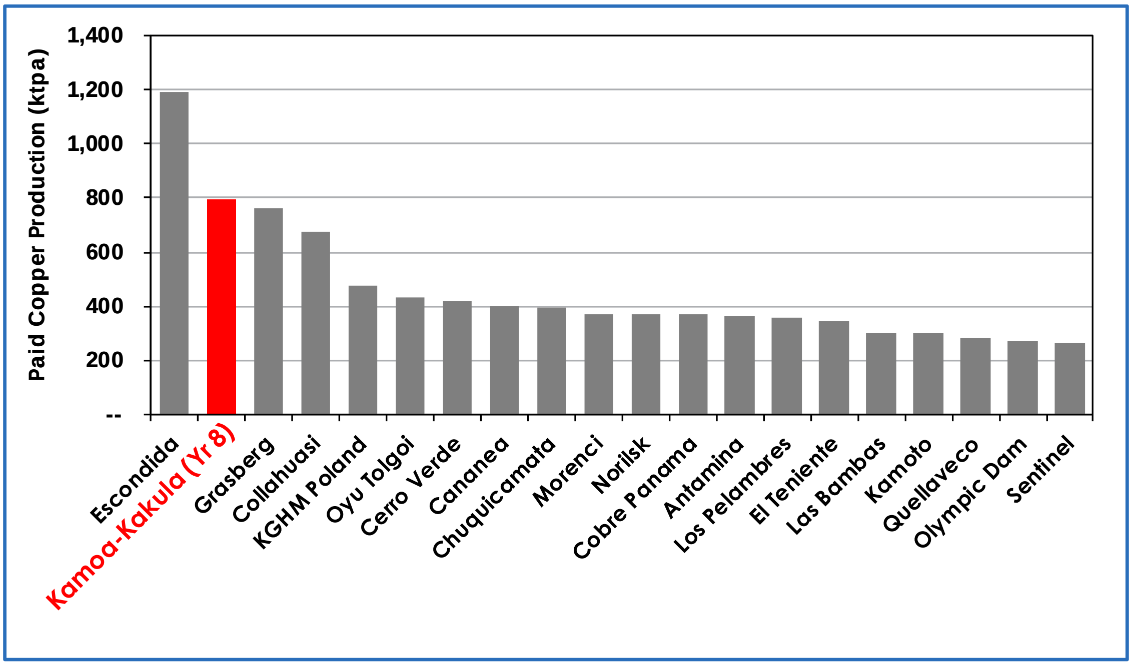


Figure by Ivanhoe, 2020. Source: Wood Mackenzie.

Figure 24.8 World Copper Producer Copper Production and Head Grade

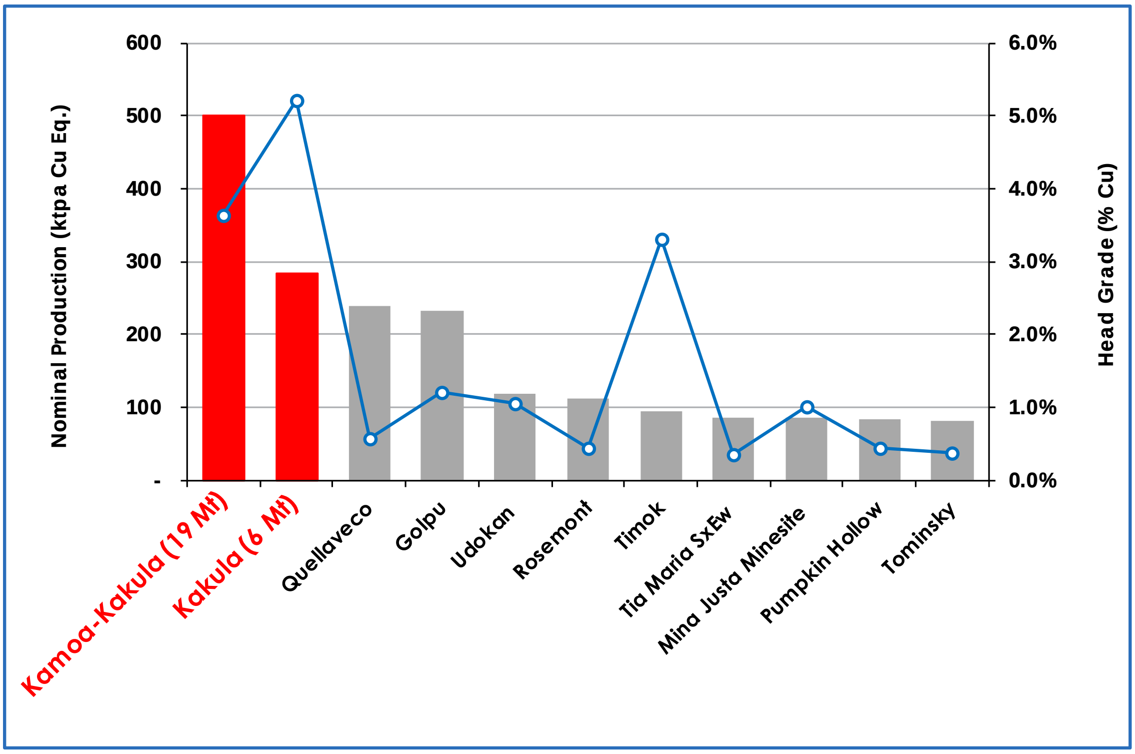


Figure by Ivanhoe, 2020. Source: Wood Mackenzie.

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 2020 PEA Revenue and Operating Costs

	Total LOM (US$M)	Years 1–5	Years 1–10	LOM Average
		(US$/t) Milled		
Revenue				
Copper in Blister	85,430	101.06	151.08	142.95
Copper in Concentrate	39,713	237.97	143.89	66.45
Acid Production	4,930	4.38	8.23	8.25
Gross Sales Revenue	130,074	343.41	303.20	217.65
Less: Realisation Costs				
Transport	9,132	31.62	23.03	15.28
Treatment and Refining	2,941	10.68	7.83	4.92
Royalties and Export Tax	7,038	21.00	16.74	11.78
Total Realisation Costs	19,111	63.30	47.60	31.98
Net Sales Revenue	110,963	280.11	255.60	185.67
Site Operating Costs				
Underground Mining	21,070	39.57	35.26	35.26
Processing	8,811	13.82	14.02	14.74
Tailings	67	0.26	0.15	0.11
Smelter	4,464	4.58	7.48	7.47
General and Administration	1,767	8.30	5.28	2.96
SNEL Discount	–292	–3.43	–2.57	–0.49
Customs Duties	1,426	2.55	2.48	2.39
Total	37,313	65.66	62.10	62.44
Net Operating Margin	73,650	214.45	193.50	123.24
Net Operating Margin (%)	66.37%	76.56%	75.71%	66.37%

Table 24.6 Kamoa-Kakula 2020 PEA Capital Costs

Capital Costs (US$M)	Initial Capital (US$M)	Expansion Capital (US$M)	Sustaining Capital (US$M)	Total (US$M)
Underground Mining				
Underground Mining	156	995	5,620	6,772
Mining Infrastructure and Mobile Equipment	68	299	2,251	2,618
Capitalised Pre-Production	78	–	–	78
Subtotal	302	1,295	7,872	9,468
Power and Smelter				
Smelter Total	–	635	368	1,003
Power Supply Off Site	36	–	–	36
Subtotal	36	635	368	1,039
Concentrator and Tailings				
Plant	124	646	345	1,115
Tailings	15	26	550	591
Subtotal	139	672	895	1,706
Infrastructure				
Plant Infrastructure	69	536	678	1,283
Other Infrastructure	–	353	145	498
Overland Conveyors	–	118	66	183
Rail	–	72	84	156
Subtotal	69	1,079	973	2,120
Indirects				
EPCM	37	111	35	184
Owners Cost	70	63	–	132
Customs Duties	9	137	361	507
Closure	–	–	308	308
Subtotal	116	311	704	1,130
Capital Expenditure Before Contingency	661	3,991	10,811	15,464
Contingency	53	470	1,147	1,670
Capital Expenditure After Contingency	715	4,461	11,958	17,134

The after-tax NPV sensitivity to metal price variation is shown in Table 24.7 for copper prices from US$2.00–US$4.50/lb. The net cash flow is tabulated in Table 24.8.

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 2020 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**	**4.50**
Undiscounted	8,839	21,888	35,185	37,844	48,517	60,961	70,509
4.0%	4,620	11,251	18,056	19,416	24,876	31,243	36,089
6.0%	3,351	8,357	13,495	14,520	18,640	23,446	27,089
8.0%	2,422	6,323	10,320	11,117	14,318	18,054	20,876
10.0%	1,733	4,855	8,046	8,681	11,231	14,210	16,453
12.0%	1,213	3,771	6,374	6,891	8,968	11,393	13,214
15.0%	655	2,619	4,603	4,996	6,573	8,416	9,794
IRR (%)	21.1	37.8	53.2	56.2	67.3	79.9	89.0

Figure 24.9 Kamoa-Kakula 2020 PEA Projected Cumulative Cash Flow

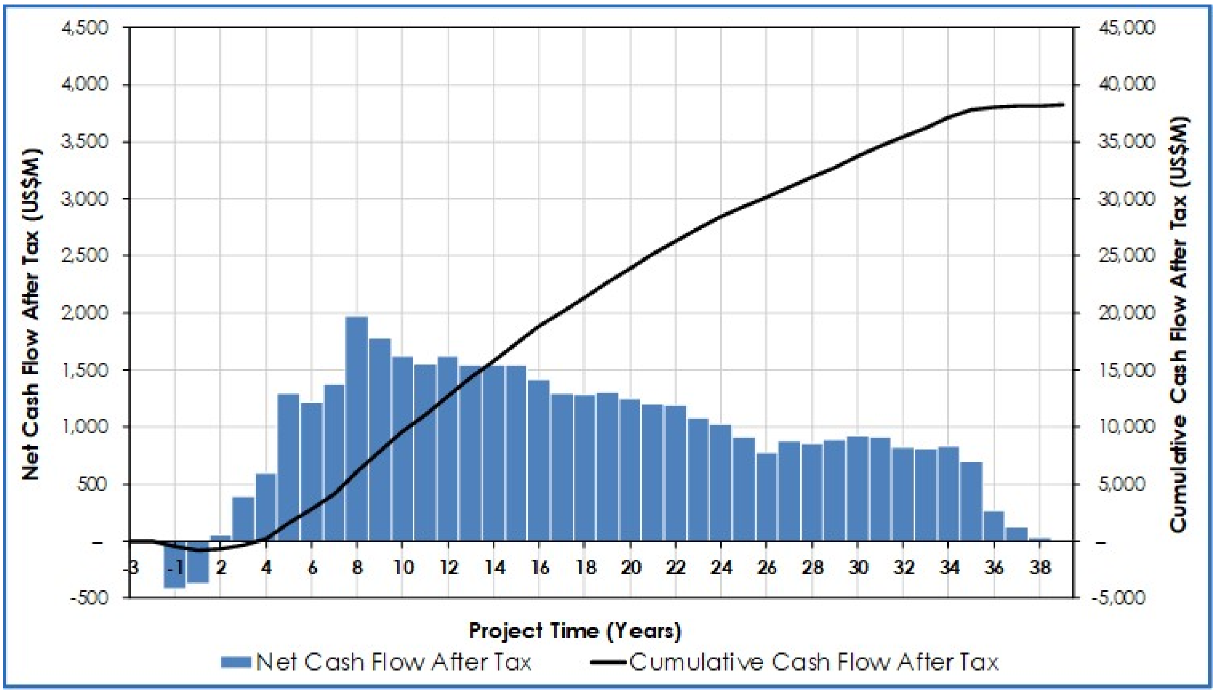


Figure by OreWin, 2020.

Table 24.8 Cash Flow – Kamoa-Kakula 2020 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	130,074	–	–	556	1,321	2,391	2,805	3,620	23,853	42,889	52,640
Realisation Costs	19,110	–	–	119	272	527	620	433	3,453	5,795	7,892
Net Revenue	110,964	–	–	437	1,049	1,865	2,185	3,186	20,400	37,095	44,748
Operating Costs											
Mining	21,070	–	–	97	192	262	313	368	2,786	6,897	10,155
Processing	8,811	–	–	30	61	94	87	158	1,167	2,784	4,430
Tailings	67	–	–	1	2	2	2	2	9	21	30
Smelter	4,464	–	–	–	–	–	–	143	710	1,503	2,109
General and Administration	1,767	–	–	18	37	65	69	69	343	648	518
Discount on Power	–292	–	–	–5	–10	–17	–20	–54	–186	–	–
Customs Duties	1,426	–	–	5	11	16	18	30	203	464	680
Total Operating Costs	37,313	–	–	147	293	422	468	715	5,030	12,317	17,921
Operating Surplus / (Deficit)	73,651	–	–	290	756	1,443	1,717	2,471	15,370	24,778	26,827
Capital Costs											
Initial Capital	715	–	386	329	–	–	–	–	–	–	–
Expansion Capital	4,461	–	6	237	558	789	675	308	938	64	885
Sustaining Capital	11,958	–	–	10	38	96	342	275	2,203	3,677	5,317
Working Capital	–	–	–	–41	–57	–79	–31	–70	–99	90	287
VAT	46	–	–23	–36	–38	–57	145	75	–	–	–20
Net Cash Flow Before Tax	56,564	–	–415	–363	65	422	814	1,893	12,130	21,127	20,891
Income Tax	18,720	–	–	6	13	24	210	592	4,141	6,760	6,974
Net Cash Flow After Tax	37,844	–	–415	–369	51	398	604	1,301	7,989	14,367	13,917

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Table 24.9 Processing Production Schedule – Kamoa-Kakula 2020 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	597,621	1,536	3,800	7,200	7,600	11,000	11,400	14,800	18,600	19,000	19,000	19,000	19,000	19,000	19,000	19,000
Cu Feed Grade	% Cu	3.63	6.83	6.13	5.89	6.47	5.45	5.47	4.86	5.02	4.60	4.40	4.23	4.31	4.13	4.02	3.97
Fe Feed Grade	% Fe	5.54	4.51	4.60	4.96	4.92	5.36	5.57	5.82	5.43	5.35	5.29	5.30	5.38	5.35	5.34	5.34
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	1.90	1.84	1.60	1.79	1.97	2.08	2.23	2.38	2.19	1.97	1.75	1.77	1.88	1.83	1.70	1.72
Copper Conc. Produced	kt (dry)	42,818	154	348	676	796	1,089	1,139	1,413	1,727	1,594	1,498	1,433	1,456	1,406	1,362	1,351
Copper Conc. - External Smelter	kt (dry)	11,944	154	348	676	796	89	139	413	727	594	498	433	456	406	362	351
Copper Conc. - Internal Smelter	kt (dry)	30,874	–	–	–	–	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	%	86.42	84.25	85.69	85.24	86.29	86.66	86.71	87.00	86.86	86.55	86.15	86.07	85.98	85.82	85.72	85.47
Copper Concentrate Grade	% Cu	43.76	57.32	57.32	53.49	53.29	47.73	47.47	44.26	46.97	47.47	48.13	48.22	48.34	47.84	48.09	47.76
Contained Copper in Conc. - External Smelter	Mlb	13,251	195	440	797	935	112	175	522	918	751	629	547	576	513	457	443
Contained Copper in Conc. - External Smelter	kt	6,010	88	200	362	424	51	79	237	417	341	286	248	261	233	207	201
Contained Copper in Blister - Internal Smelter	Mlb	27,641	–	–	–	–	1,018	1,001	844	857	904	946	961	961	955	972	964
Contained Copper in Blister - Internal Smelter	kt	12,538	–	–	–	–	462	454	383	389	410	429	436	436	433	441	437
Total Recovered Copper Production	Mlb	40,892	195	440	797	935	1,130	1,176	1,366	1,775	1,654	1,575	1,509	1,537	1,468	1,429	1,408
Total Recovered Copper Production	kt	18,548	88	200	362	424	513	534	620	805	750	714	684	697	666	648	638

Table 24.10 Processing Production Schedule – Kamoa-Kakula 2020 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	19,000	19,000	19,000	19,000	19,000	19,000	19,000	19,000	19,000	19,000	19,000	19,000	19,000	19,000	19,000	19,000
Cu Feed Grade	% Cu	3.71	3.55	3.46	3.48	3.32	3.34	3.34	3.34	3.33	3.08	2.98	2.90	2.83	2.86	2.83	2.75
Fe Feed Grade	% Fe	5.28	5.35	5.30	5.43	5.48	5.55	5.59	5.77	5.77	5.61	5.61	5.60	5.59	5.60	5.61	5.66
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	1.60	1.60	1.64	1.70	1.75	1.61	1.84	2.24	2.38	1.90	1.87	1.85	1.87	1.92	1.96	1.81
Copper Conc. Produced	kt (dry)	1,271	1,195	1,145	1,201	1,186	1,219	1,350	1,431	1,441	1,352	1,312	1,274	1,242	1,259	1,228	1,189
Copper Conc. - External Smelter	kt (dry)	271	195	145	201	186	219	350	431	441	352	312	274	242	259	228	189
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	%	85.37	84.80	84.47	85.16	85.45	85.86	86.81	87.64	88.14	87.65	87.20	87.28	87.46	87.52	87.46	87.30
Copper Concentrate Grade	% Cu	47.36	47.93	48.47	46.91	45.42	44.65	40.84	38.92	38.68	37.90	37.66	37.72	37.83	37.73	38.24	38.37
Contained Copper in Conc. - External Smelter	Mlb	343	247	183	254	234	276	376	382	394	297	257	228	201	214	189	156
Contained Copper in Conc. - External Smelter	kt	155	112	83	115	106	125	171	173	179	135	117	104	91	97	86	71
Contained Copper in Blister - Internal Smelter	Mlb	970	1,001	1,025	973	938	909	827	833	823	821	819	819	822	821	834	837
Contained Copper in Blister - Internal Smelter	kt	440	454	465	441	426	412	375	378	373	372	372	371	373	373	378	380
Total Recovered Copper Production	Mlb	1,313	1,248	1,208	1,227	1,173	1,186	1,203	1,215	1,216	1,117	1,077	1,047	1,023	1,035	1,022	993
Total Recovered Copper Production	kt	595	566	548	557	532	538	546	551	552	507	488	475	464	469	464	450

Table 24.11 Processing Production Schedule – Kamoa-Kakula 2020 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	19,000	19,000	16,040	14,111	7,218	4,000	1,291	1,000	1,000	1,000	1,000	25	–	–	–	–
Cu Feed Grade	% Cu	2.63	2.59	2.57	2.48	2.55	2.86	3.17	3.19	3.06	2.86	2.80	2.80	–	–	–	–
Fe Feed Grade	% Fe	5.42	5.61	6.00	6.71	6.55	7.19	6.29	5.90	5.96	5.81	5.84	5.84	–	–	–	–
As Feed Grade	% As	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.01	0.01	0.01	–	–	–	–
S Feed Grade	% S	1.70	1.75	1.88	2.25	2.68	3.15	2.70	2.32	2.24	1.75	1.76	1.76	–	–	–	–
Copper Conc. Produced	kt (dry)	1,138	1,128	955	808	412	265	96	74	72	67	66	2	–	–	–	–
Copper Conc. - External Smelter	kt (dry)	138	128	–45	–66	412	265	96	74	72	67	66	2	–	–	–	–
Copper Conc. - Internal Smelter	kt (dry)	1,000	1,000	1,000	874	–	–	–	–	–	–	–	–	–	–	–	–
Copper Concentrate Recovery	%	87.02	86.93	86.89	86.65	86.85	87.53	88.11	88.13	87.91	87.52	87.40	87.40	–	–	–	–
Copper Concentrate Grade	% Cu	38.19	37.95	37.51	37.45	38.74	37.76	37.53	37.91	37.41	37.23	37.07	37.07	–	–	–	–
Contained Copper in Conc. - External Smelter	Mlb	113	104	–37	–55	352	220	80	62	59	55	54	1	–	–	–	–
Contained Copper in Conc. - External Smelter	kt	51	47	–17	–25	160	100	36	28	27	25	24	1	–	–	–	–
Contained Copper in Blister - Internal Smelter	Mlb	833	827	814	711	–	–	–	–	–	–	–	–	–	–	–	–
Contained Copper in Blister - Internal Smelter	kt	378	375	369	323	–	–	–	–	–	–	–	–	–	–	–	–
Total Recovered Copper Production	Mlb	946	931	777	657	352	220	80	62	59	55	54	1	–	–	–	–
Total Recovered Copper Production	kt	429	422	353	298	160	100	36	28	27	25	24	1	–	–	–	–

24.4 Kamoa-Kakula 2020 PEA Mining

The Kamoa-Kakula 2020 PEA analyses a production case with an expansion of the Kakula concentrator processing facilities, and associated infrastructure to 19 Mtpa and includes a smelter and eight separate underground mining operations with associated capital and operating costs. The locations of the eight mines and the boundaries for the PFS and PEA cases are shown in Figure 24.1. The eight mines ranked by their relative net present values are:

- Kakula Mine (PFS 6.0 Mtpa).
- Kansoko Mine (PFS 1.6 Mtpa to 6.0 Mtpa).
- Kakula West Mine (PEA 6.0 Mtpa).
- Kamoa North Mine 1 (PEA 6.0 Mtpa).
- Kamoa North Mine 2 (PEA 6.0 Mtpa).
- Kamoa North Mine 3 (PEA 6.0 Mtpa).
- Kamoa North Mine 4 (PEA 3.0 Mtpa).
- Kamoa North Mine 5 (PEA 1.0 Mtpa).

Mining methods in the Kamoa-Kakula 2020 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 2020 FS, 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 IDP20.

Figure 24.10 Kamoa-Kakula 2020 PEA Development and Mining Zones

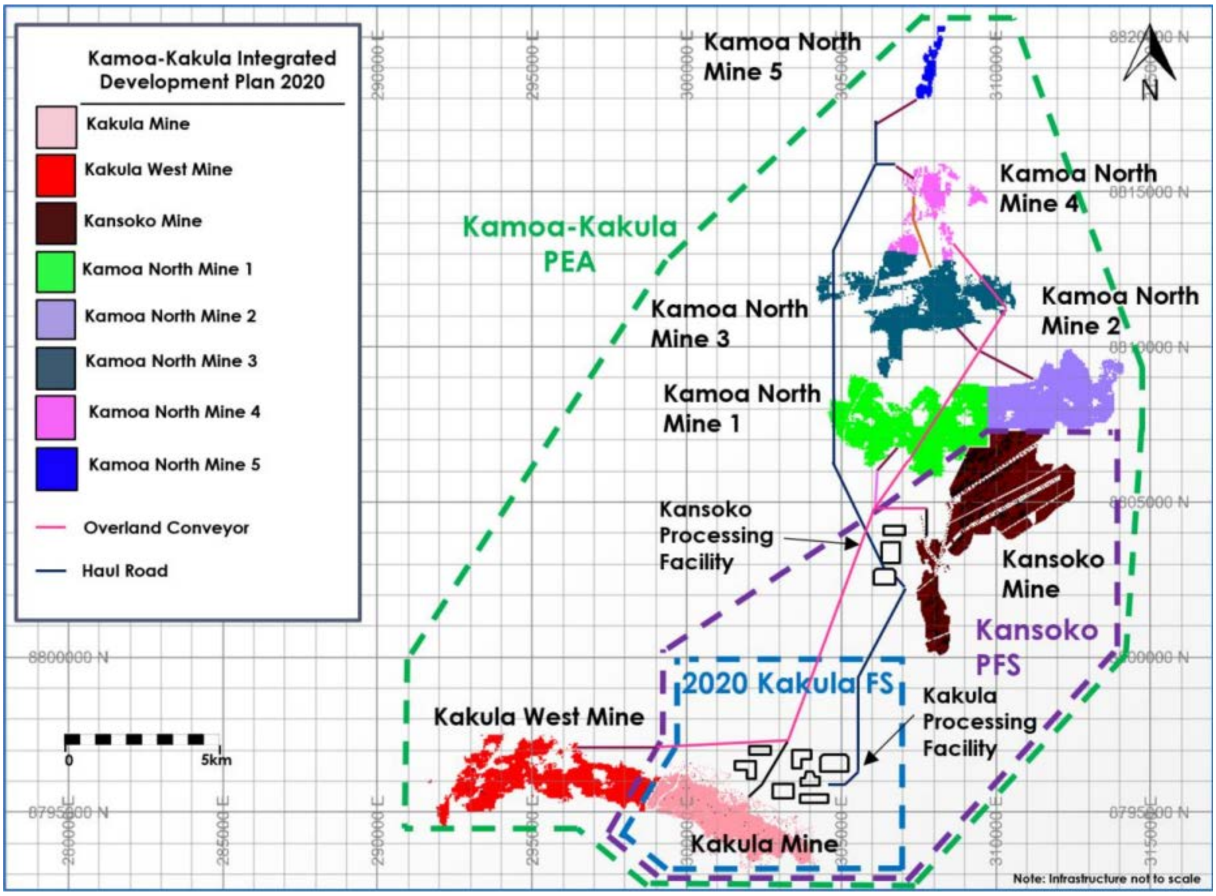


Figure by OreWin, 2020.

24.4.1 Kakula West Mining

The West Scarp Fault was used to define the Kakula West resource model from the Kakula resource model. Figure 24.11 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 hanging wall 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 2020 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 (3–6 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 24.12.

Figure 24.11 Kakula West Resource Model Area

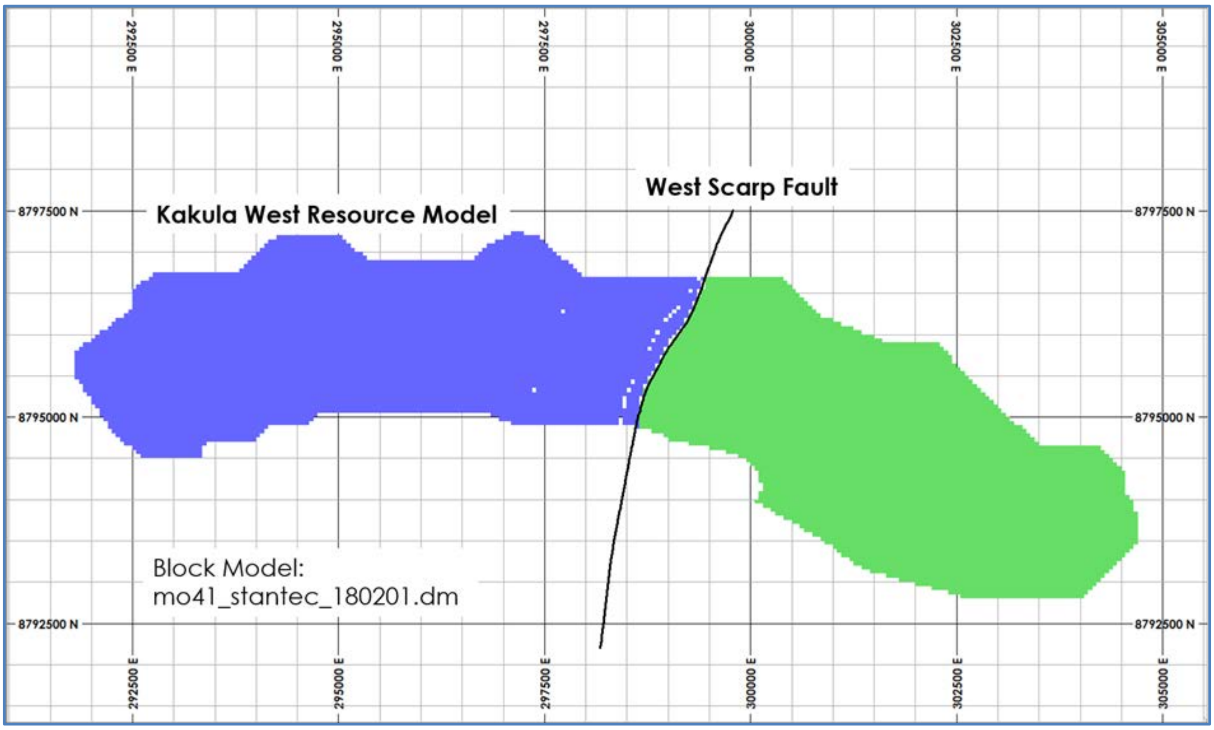


Figure by OreWin, 2019.

Table 24.12 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 x 2.20 km and the depth of the Kakula West targeted resource is between 270–1,000 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. 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 24.12. 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.

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 east–west 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 development design and mining zones are shown in Figure 24.12. The ventilation raise and shaft locations are shown in Figure 24.13.

Figure 24.12 Kakula West 2020 PEA Development

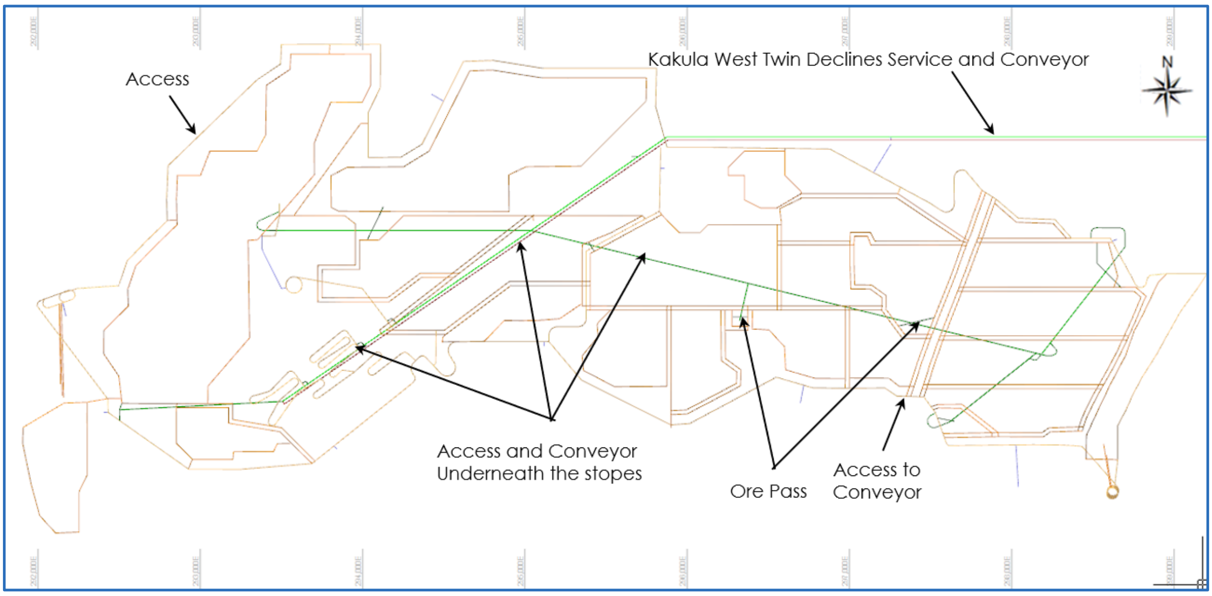


Figure by OreWin, 2020.

Figure 24.13 Kakula West 2020 PEA Ventilation Location

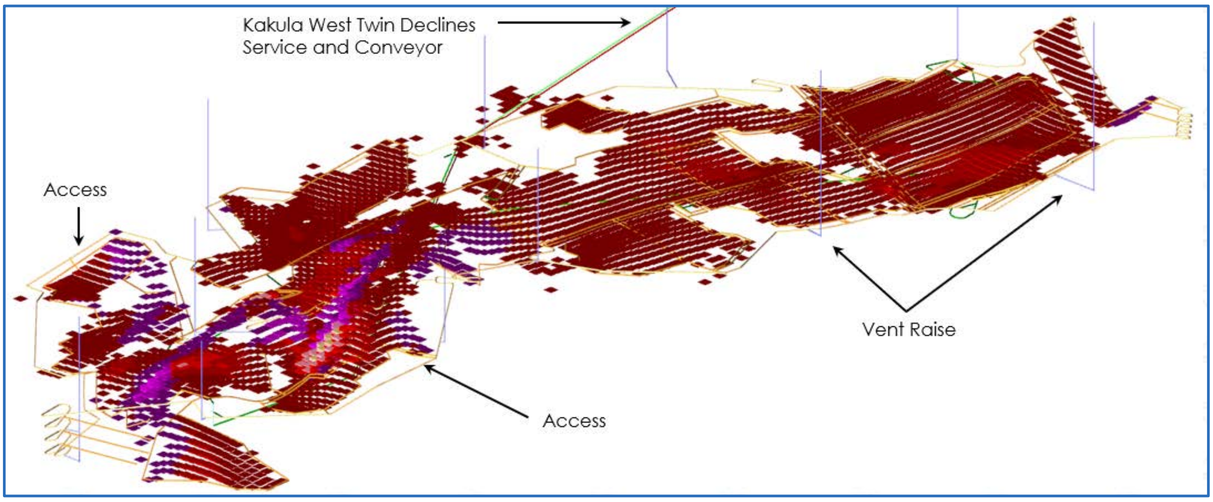


Figure by OreWin, 2020.

A Kakula West Mineral Resource of approximately 85.9 Mt at 3.80% Cu has been defined on multiple mining zones. The mine production rate is 6.0 Mtpa and the LOM is 18-years. The 6.0 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 Kakula 2020 FS.

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 $90/t NSR21 cut-off grade. The Resources model name is mk50ton.dm. The study assesses the development and production of the Kamoa North deposit at a maximum of 19 Mtpa underground mine production from five separate mines. The locations of the five Kamoa North mines and the boundaries for the PFS and PEA cases are shown in Figure 24.14. The five Kamoa North mines ranked by their relative values are:

- Kamoa North Mine 1 (PEA 6.0 Mtpa).
- Kamoa North Mine 2 (PEA 6.0 Mtpa).
- Kamoa North Mine 3 (PEA 6.0 Mtpa).
- Kamoa North Mine 4 (PEA 3.0 Mtpa).
- Kamoa North Mine 5 (PEA 1.0 Mtpa).

Figure 24.14 Kamoa-Kakula 2020 PEA Mining Locations

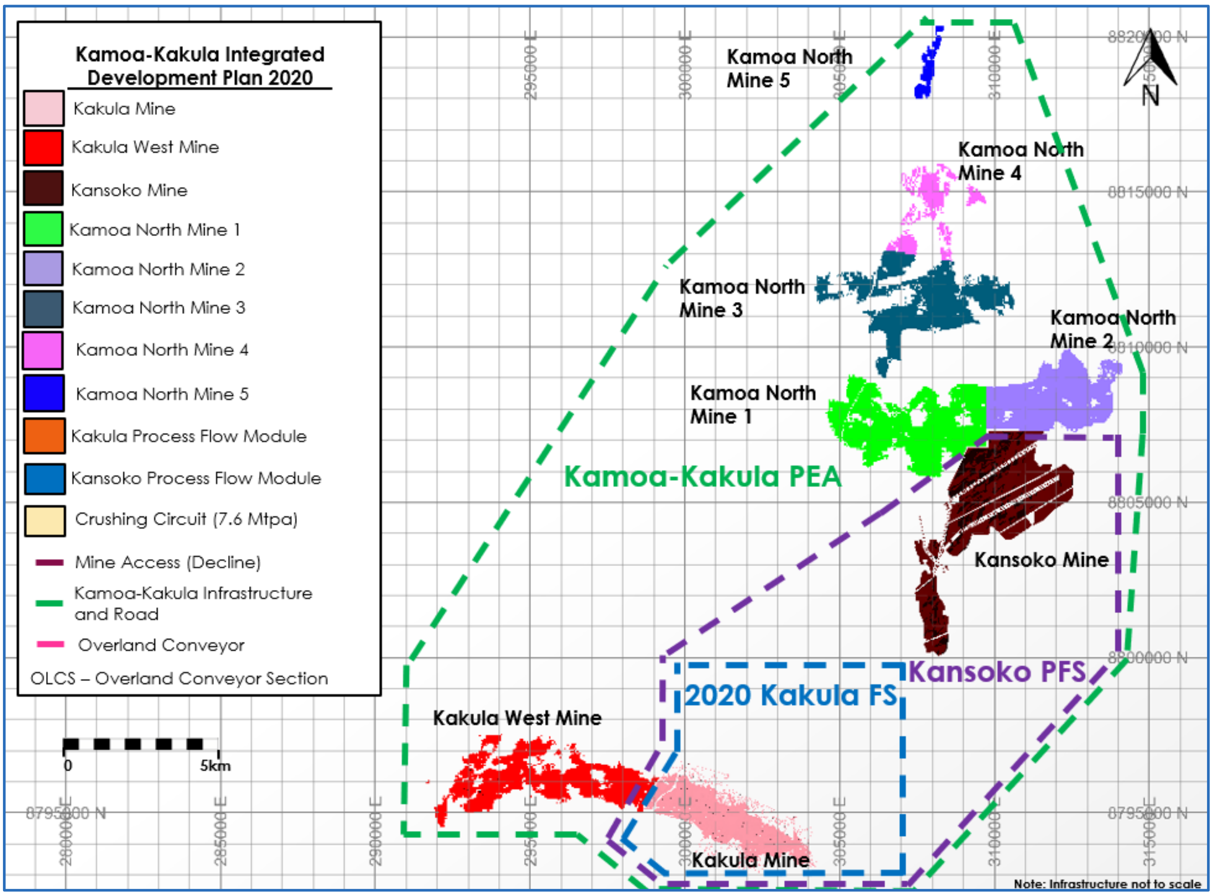


Figure by OreWin, 2020.

The Kamoa North mines are based on a preliminary UG optimisation at $90/t NSR cut-off grade. The Resources model name is mk50ton.dm.

Stope optimisation was undertaken on the resource model at mining cut-off grades of $90/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.

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 Kakula 2020 FS 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 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, 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.

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 North Mine 1 and Kamoa North 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 2020 FS.

The Kamoa North optimisation, development and production need to be further investigated as a part of future studies. The metres of the required lateral and vertical development to access the mining zones were estimated for each mine separately as shown in Table 24.13.

Table 24.13 Kamoa North Mines Development Required

	Unit	Kamoa North Mine 1	Kamoa North Mine 2	Kamoa North Mine 3	Kamoa North Mine 4	Kamoa North Mine 5
Mine Production Rate	Mtpa	6.0 Mtpa	6.0 Mtpa	6.0 Mtpa	3.0 Mtpa	1.0 Mtpa
Lateral Development						
Conveyor Decline	m	1,725	6,432	2,240	2,035	1,586
Lateral Development	m	163,424	97,188	112,367	48,617	15,647
Total Lateral Development	m	165,150	103,620	114,607	50,652	17,233
Vertical Development						
Ventilation Raise	m	2,571	5,542	2,961	1,761	730
Ore pass	m	327	258	226	125	–
Total Vertical Development	m	2,898	5,801	3,186	1,886	730

24.6 Kamoa-Kakula 2020 PEA Processing and Infrastructure

The Kamoa-Kakula 2020 PEA assesses an alternative development option of mining several deposits on the Kamoa-Kakula Project as an integrated, 19 Mtpa mining, processing and smelting complex, built in three stages. This scenario envisages the construction and operation of three separate mines: first, an initial 6.0 Mtpa mining operation would be established at the Kakula Mine on the Kakula Deposit; this is followed by a subsequent, separate 6.0 Mtpa mining operation at the Kansoko Mine using the existing twin declines that were completed in 2017; a third 6.0 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 five other mines in the Kamoa North area to maintain throughput of 19 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 and Kansoko. 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.

The annual production results including processing, concentrate and smelter production are shown in Table 24.9 to Table 24.11.

The 19 Mtpa scenario is comprised of five processing plants spread between the Kamoa and Kansoko sites. These processing plants are configured in a combination of Kakula and Kansoko process flows.

Processing Summary:

- Total processing capacity is 19 Mtpa
- Kakula plant site (4 x 3.8 Mtpa)
  - 3 x Kakula process plants
  - 1 x Kansoko process plant
- Kansoko plant site (1 x 3.8 Mtpa)
  - 1 x Kansoko process plant

In total there will be three crushing circuits, located at Kakula plant site, Kansoko plant site and Kamoa North. The crushers will be relocated as required during the LOM.

Crushing Circuits Summary:

- Crusher 1: Kakula plant site (1 x 7.6 Mtpa)
  - Installed in Year-1
  - Relocated to Kamoa North in Year-25
- Crusher 2: Kansoko plant site (1 x 7.6 Mtpa)
  - Installed in Year-7
- Crusher 3: Kakula plant site (1 x 7.6 Mtpa)
  - Installed in Year-10
  - Relocated to Kansoko plant site in Year-24
  - Relocated to Kamoa North in Year-30

Overland Conveyors Sections 1–4 (OLCS 1–4) from Kansoko and Kamoa North mines will be used to convey ore to the crushing circuits and mills.

Overland Conveyor Sections Summary:

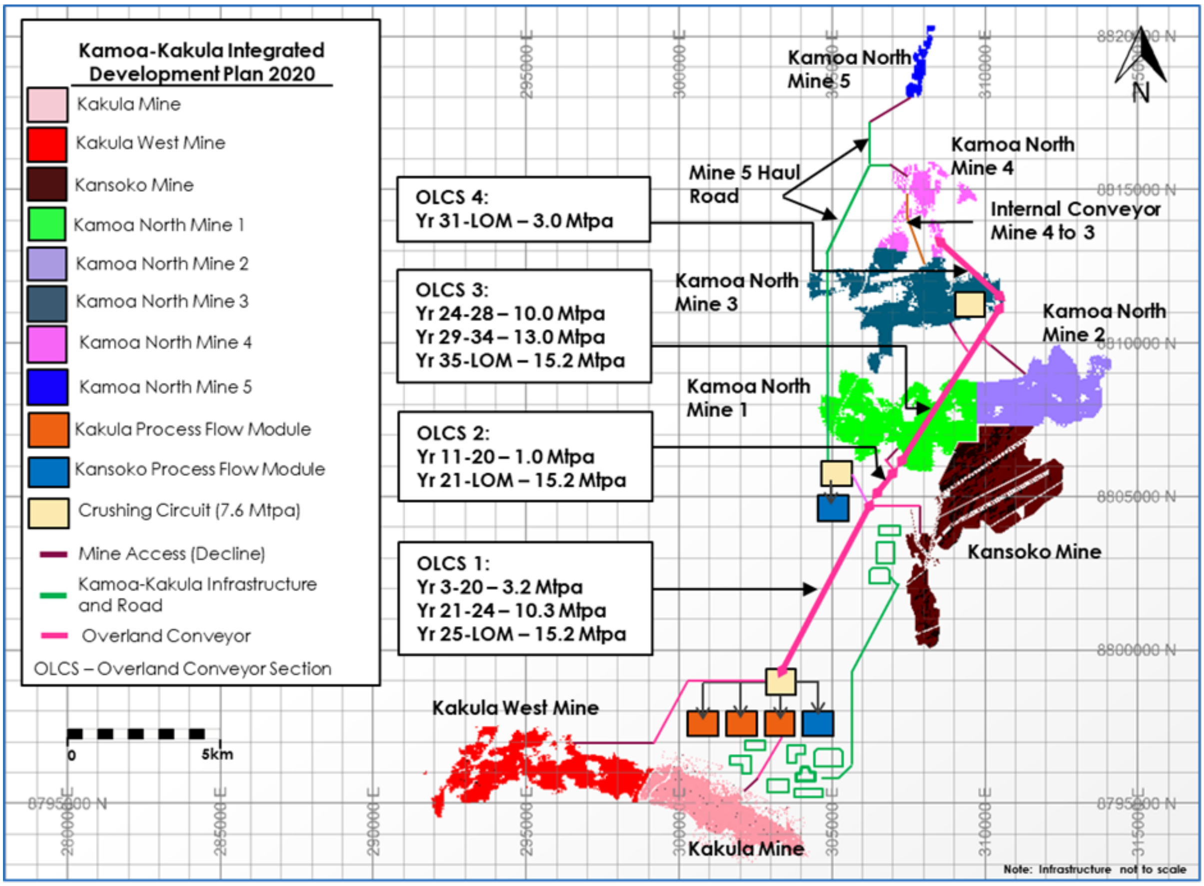
- OLSC 1 Capacity:
  - Year-5 to Year–20 – 3.2 Mtpa
  - Year-21 to Year-24 – 10.3 Mtpa
  - Year-25 to LOM – 15.2 Mtpa
- OLCS 2 Capacity:
  - Year-11 to Year-20 – 1.0 Mtpa

- Year-21 to LOM – 15.2 Mtpa

- OLCS 3 Capacity:
  - Year24 to Year-28 – 10.0 Mtpa
  - Year-29 to Year-34 – 13.0 Mtpa
  - Year-35 to LOM – 15.2 Mtpa
- OLCS 4 Capacity:
  - Year-31 to LOM – 3.0 Mtpa

Haul road to support 1.0 Mtpa production from Kamoa North mine 5 will be constructed in Year-36.

Figure 24.15 Kamoa-Kakula 2020 PEA Block Plan



24.6.1 Central Processing Facility

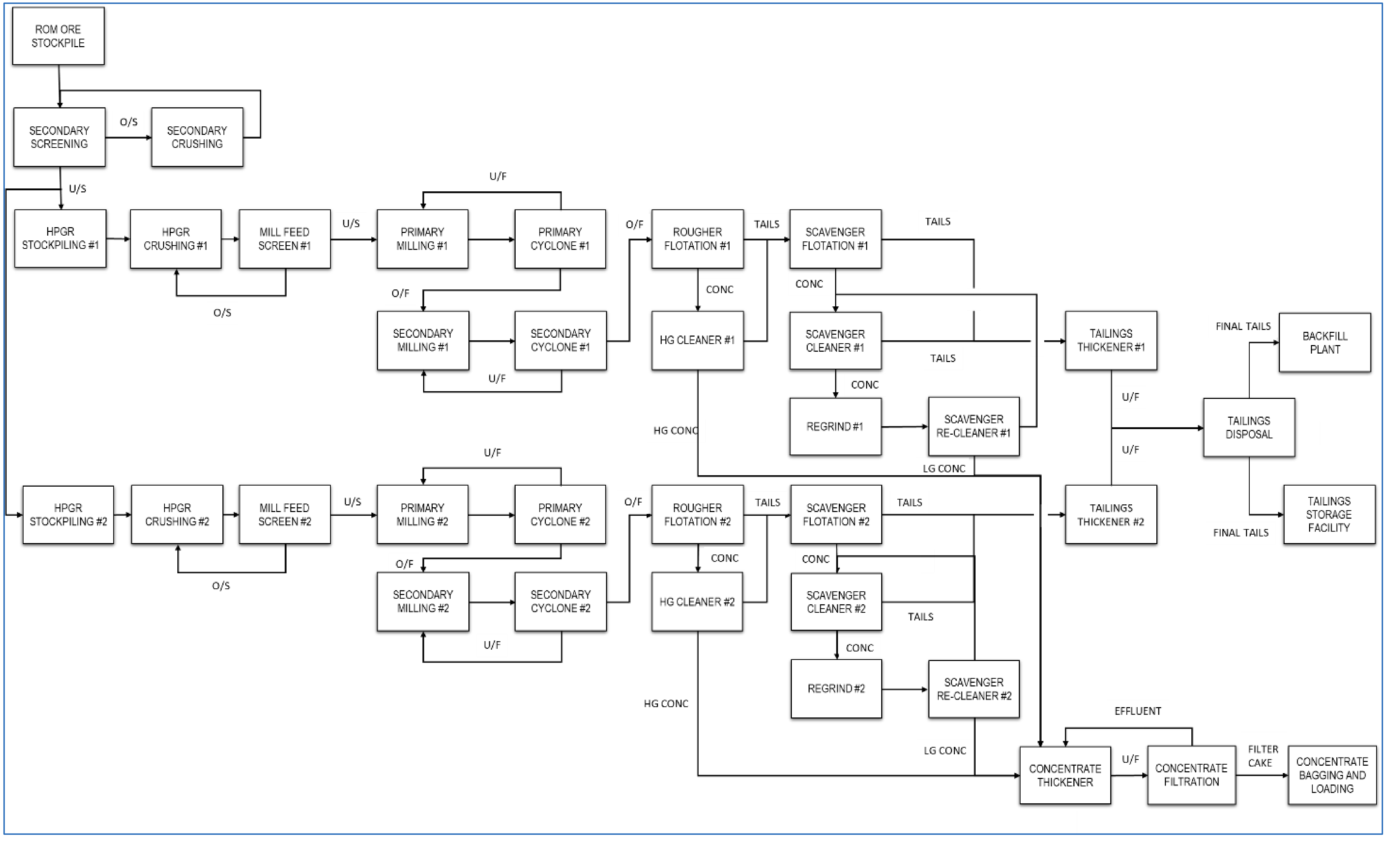
The Kakula process plant will be the first of five 3.8 Mtpa circuits to be located at the central processing complex. The Kakula concentrator (Central Complex Concentrator 1) includes a ROM stockpile to feed a 15.2 Mtpa Run-of-Mine (ROM) concentrator based on staged crushing and screening, followed by 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 3.8 Mtpa in line with the mine ramp up and the throughput will be doubled during Phase 2 to 7.6 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 24.16.

Following the ramp-up of Central Complex Concentrator 1 to 7.6 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 7.6 Mtpa to 15.2 Mtpa will also be completed in a two phased approach, as dictated by the mining plan.

Kansoko ROM material will be stored on the 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 6 km Kansoko Overland Conveyor 1. The Kansoko Overland Conveyor Section 1 (OLCS1) will transport the material the 11 km from Kansoko Overland Conveyor 2, which will transfer the Kansoko material to the Central Complex Concentrator 2 ROM stockpile. Central Complex Concentrator 2 also consists of a 7.6 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.

Figure 24.16 Central Complex Concentrator 1 Block Flow Diagram



A high level block flow diagram of the Central Complex Concentrator 2 circuit is shown in Figure 24.17. Once Central Complex Concentrator 2 has ramped up to 7.6 Mtpa, the complex will be expanded by the addition of the Central Complex Concentrator 3 at the Kakula Mine Area. Central Complex Concentrator 3 at Kansoko will also be based on the Kansoko circuit design (Figure 24.17).

Figure 24.17 Central Complex Concentrator 2 and 3 Block Flow Diagram

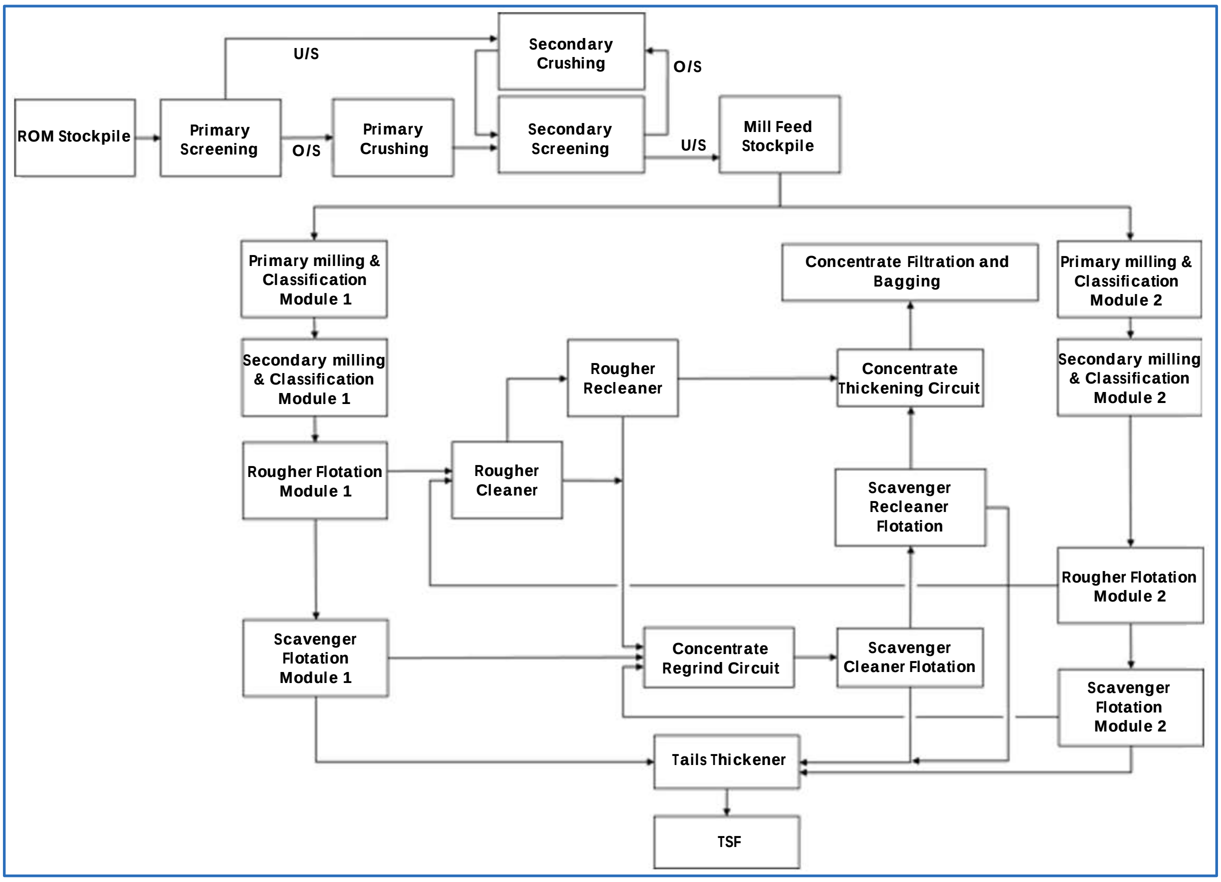


Figure by DRA, 2018

Kakula West ROM material will be stored on the 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.

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 online to supplement tonnage to the central complex. Kamoa North Mine 1 ROM material will be stored on the 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 19 Mtpa processing rate.

Kamoa North Mine 2 ROM material will be stored on the 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 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 19 Mtpa processing rate. Kamoa North Mine 4 ROM material will be stored on the 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 2020 PEA Smelter

Concentrate will be conveyed from the adjacent concentrator complex into a concentrate shed located in the smelter complex. The smelting process utilises the direct-to-blister smelting technology (DBF), which is proven for treating high copper, low sulfur copper concentrates similar to those envisaged for the Kamoa-Kakula project. Copper concentrate is first dried in a steam dryer before being oxidised with oxygen enriched air in the reaction shaft of the DBF to produce blister copper and SO_2 off gas in a single stage flash smelting process. The SO_2 laden off gases are dedusted and sent to a double-contact-double adsorption sulfuric 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 transported 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 sulfuric acid. The flow sheet schematic of the DBF process is shown in Figure 24.18. The smelter feed copper:sulfur 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 24.18 Flow Sheet Schematic of the Direct-to-Blister Smelting Process

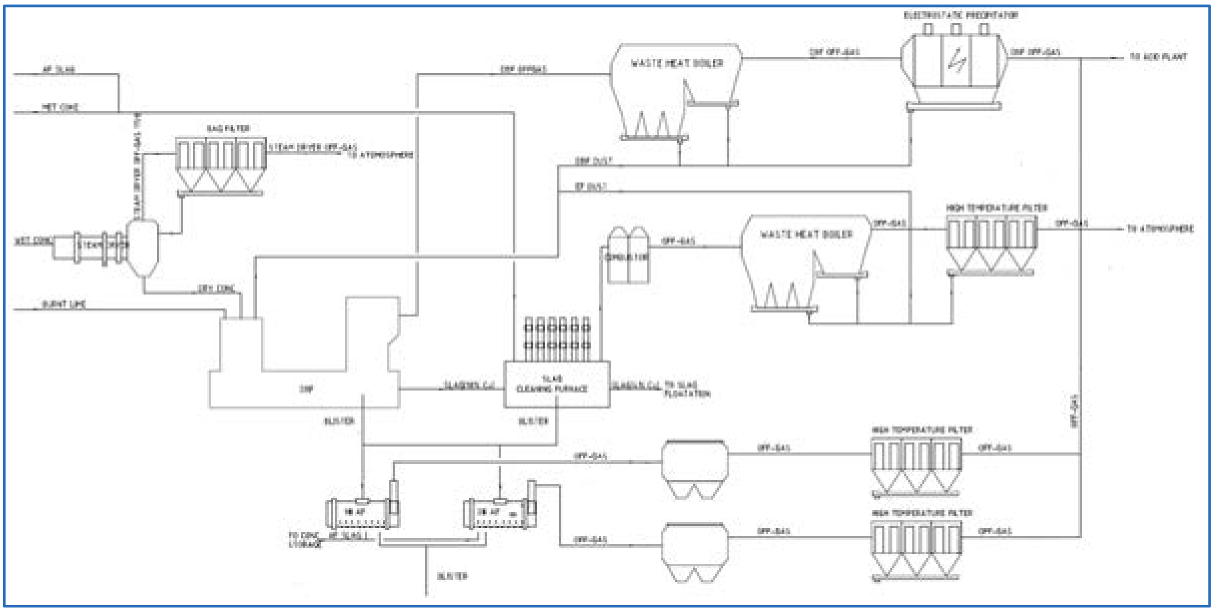


Figure by Nerin, 2018.

24.6.3 Kamoa-Kakula 2020 PEA Infrastructure

The infrastructure for the Kamoa-Kakula 19 Mtpa PEA must support five, 3.8 Mtpa 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 supply will be ramped up as needed to meet the production requirements.

24.6.3.2 TSF

The Kamoa-Kakula 19 Mtpa PEA considered the expansion of the Kakula TSF to allow for the additional tailings tonnage.

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 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 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 employees will be provided by an outside contracting service.

Permanent villages called the Owners Camps, capable of accommodating 1,500 persons each, will be constructed at the mine locations to provide accommodation for owner's team management, senior staff, 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 2020 PEA Process and Infrastructure Costs

The Kamoa-Kakula 2020 PEA analysed a five-phased expansion of production from 3.8 Mtpa to 19 Mtpa (in 3.8 Mtpa increments). The following inputs and documents were identified and used in compiling the capital cost estimate:

- Kakula FS capital cost estimate (as detailed in Section 21).
- Kakula-Kansoko PFS capital cost estimate (as detailed in Section 21).
- 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 campaigns. It is recommended that the flotation response of the Kamoa North deposits be tested on the Kakula FS and Kansoko flow sheets.

During the infrastructure planning for the Kamoa-Kakula 2020 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 2020 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 2020 FS. 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 and 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). Wood has checked the data used to construct the resource model, the methodology used to construct them (Datamine macros) and has validated the resource model. Wood 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 and mining assumptions.
  - 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 development 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 2020

The Kamoa Kakula IDP20 includes an update of the Kamoa-Kakula (Kakula and Kansoko) Mineral Reserve and updates of the preliminary economic assessment (PEA) including analysis of the Kakula West and Kamoa North Mineral Resource.

The Kakula 2020 FS has identified a Mineral Reserve and development path that has confirmed significant value in the Kakula Deposit.

The Kakula-Kansoko 2020 PFS indicates the combined Kakula and Kansoko mine plans using the same 7.6 Mtpa processing facility of the Kakula 2020 FS generates significant value.

The analysis in the Kamoa Kakula 2020 PEA indicates that the potential development scenarios for the Kamoa Kakula project could include expansions and on site smelting that could provide potential additional project values, Considerable ongoing and additional studies need to be undertaken to define the development sequence and production rates including mining methods, plant sizing and location for the deposits and identify the project potential.

25.3 Mineral Reserve Estimate

25.3.1 Kakula Mineral Reserve Estimate

Mineral Reserves for the Kakula 2020 FS 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.
- As with any study as more definition of the orebody and geotechnical criteria is made available plans should be adjusted accordingly. The main criteria to be assessed for the Kakula deposit is orebody extents and elevations, ground conditions and geotechnical information, and water inflows and hydrology. Based on increased definition and confidence in these parameters the designs from the FS should be adjusted to align with the new or more detailed criteria. As a result of this definition it may be possible to move the perimeter drifts in closer to the ore body to reduce the amount of waste rock and increase the ore production in the early stages. Again it is important this information is based on the most current criteria, and analysis continues throughout the development of the orebody, as this opportunity does increase the risk if the orebody expands or the geotechnical conditions worsen requiring larger protection pillars.

25.3.2 Kamoa Mineral Reserve Estimate

Mineral Reserves for the Kamoa-Kakula IDP20 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.
- The areas previously identified for the Kakula 2020 FS stated in Section 25.3.1.

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 variability testwork and mini plot plant conducted on the Kakula material as part of the Kakula 2020 FS.

25.5 Infrastructure

Surface infrastructure construction contractors are currently established or in the process of site establishment. The general working environment is well understood with constraints of services and access to site already mitigated. This is an excellent base/benchmark on which the FS study was based on for the Kakula FS. This provides high confidence in estimating rates and construction durations used across the FS.

Bulk power supply design and construction progressed sufficiently to ensure there is a high confidence in the bulk power supply design and costs. Agreements between Kamoa and SNEL are in place to ensure a steady power supply to the operations.

There is a level of uncertainty regarding the electricity tariff used in the FS, due to the pending status of an agreed tariff between Kamoa and SNEL. The Electricity rate used for the FS is regarded as a good long-term rate.

Material transport / logistics to site for operations and construction is well understood due to the current construction and operational activities. Rates and durations from reputable logistics companies were used in the FS.

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 2020 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 Kansoko. 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 2020 FS has identified a Mineral Reserve and development path for the Kakula Deposit. It is recommended that studies at Kakula continue and incorporate a five year mining plan. This will include optimisation of the mine plan and monitoring of actuals against the budgets and design needs to be undertaken as the mine moves from development to production.
- The Kakula-Kansoko 2020 PFS has identified a Mineral Reserve. It is recommended that studies at Kansoko be progressed to feasibility study. The Kansoko study work needs to be prepared for feasibility and execution. This includes detailed plans and development of systems and procedures for of the controlled convergence room-and-pillar mining method.
- The Kamoa-Kakula 2020 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.
- The three stages of the project provide a development plan. As development continues each stage of the project should be analysed and redefined. Some of the specific studies and areas for analysis are:
  - 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 adherence 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.
  - Road and 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.

- Continue the underground exploration programme at Kakula to further define the orebody and assist with the location of the perimeter and connection drifts.
- Attain first-hand information on actual mining conditions from the on-going development 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 Kamoa and Kakula, and the goal of establishing sufficient Indicated Mineral Resources to support stand-alone mining operations at Kakula, Kakula West, Kansoko, and Kamoa North has been achieved. The future drill plan at Kakula is to continue infill drilling in support of the current mine development, and to define the edges of the higher-grade material. 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. Wood has recommended further drilling of 17,000 m at a cost of approximately $2.0M.

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 Kansoko 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.
- Continue the underground exploration programme at Kakula to further define the orebody and assist with the location of the perimeter and connection drifts.
- Attain first-hand information on actual mining conditions from the on-going development and to validate design assumptions.
- Complete hydrological studies and data evaluation to better determine impacts on underground mining conditions and productivities.
- 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 determine if it can be eliminated.
- Investigate the possibility of adding sand to the tailings at Kakula to reduce the binder quantity.
- Complete a Feasibility Study on the Kansoko orebody.

As with any study, as more definition of the orebody and geotechnical criteria is made available plans should be adjusted accordingly. The main criteria to be assessed for the Kakula deposit is orebody extents and elevations, ground conditions and geotechnical information, and water inflows and hydrology. Based on increased definition and confidence in these parameters the designs from the FS should be adjusted to align with the new or more detailed criteria. As a result of this definition it may be possible to move the perimeter drifts in closer to the ore body to reduce the amount of waste rock and increase the ore production in the early stages. Again it is important this information is based on the most current criteria, and analysis continues throughout the development of the orebody, as this opportunity does increase the risk if the orebody expands or the geotechnical conditions worsen requiring larger protection pillars.

26.4 Process Plant

The following is a list of process recommendations for the Kamoa deposit:

- Kamoa Copper 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 programme 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 characterisation and flotation variability testwork programme 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 programme 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 programme.
- 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 programme.

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 2020 FS 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 2020 FS 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

It is recommended that further studies of the Kamoa-Kakula infrastructure should be undertaken.

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