OreWin
Independent Mining Consultants

KAMOA COPPER SA

Kamoa-Kakula Project

Kamoa-Kakula Integrated Development Plan 2023

March 2023

Job No. 22005

IMPORTANT NOTICE

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OreWin Pty Ltd ACN 165 722 574
140 South Terrace Adelaide 5000
P +61 8 8210 5600 **E** orewin@orewin.com **W** orewin.com

Title Page

Project Name:	Kamoa-Kakula Project
Title:	Kamoa-Kakula Integrated Development Plan 2023
Location:	Lualaba Province Democratic Republic of the Congo
Effective Date of Technical Report:	6 March 2023
Effective Date of Mineral Resources:	Kamoa: 30 January 2020 Kakula: 31 December 2022
Effective Date of Drilling Database:	Kamoa Resource: 20 January 2020 Kakula Resource: 13 December 2022
Effective Date of Mineral Reserves:	31 December 2022

Qualified Persons:

- Bernard Peters, B. Eng. (Mining), FAusIMM (201743), employed by OreWin as Technical Director - Mining was responsible for: Sections 1.1 to 1.6, 1.15 to 1.16.1, 1.16.3 to 1.17.1, 1.17.3, 1.18, 1.18.2, 1.19, 1.19.1; Section 2; Section 3; Section 4; Section 5; Section 6; Section 15; Section 16.2 to 16.3.7, 16.3.8.1, 16.3.8.2, 16.3.9, 16.4 to 16.4.7, 16.4.8.1, 16.4.8.2, 16.4.9, 16.5 to 16.5.7, 16.5.8.1, 16.5.8.2, 16.5.9, 16.6 to 16.6.7, 16.6.8.1, 16.6.8.2, 16.6.9, 16.7 to 16.7.7, 16.7.8.1, 16.7.8.2, 16.7.9; Section 19; Section 20; Sections 21.1 to 21.2.1, 21.3; Section 22; Section 23; Sections 24.1 to 24.4, 24.6; Sections 25.2, 25.3; Sections 26.1, 26.3; Section 27.
- Jeremy Witley, Pr.Sci.Nat., SACNASP, FGSSA, HOD, Mineral Resources, The MSA Group, was responsible for: Sections 1.7 to 1.11, 1.13, 1.14, 1.18.1, 1.19, 1.19.2; Section 2; 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 1.19; Section 2; Section 10.6; Section 16.1; Section 27.
- Curtis Smith, B. Eng. (Mining), MAusIMM(CP) (311458), employed by OreWin as Principal Mining Engineer, was responsible for: Sections 1.1 to 1.6, 1.15 to 1.16.1, 1.16.3 to 1.17.1, 1.17.3, 1.18, 1.18.2, 1.19, 1.19.1; Section 2; Section 3; Section 4; Section 5; Section 6; Section 15; Section 16.2 to 16.3.7, 16.3.8.1, 16.3.8.2, 16.3.9, 16.4 to 16.4.7, 16.4.8.1, 16.4.8.2, 16.4.9, 16.5 to 16.5.7, 16.5.8.1, 16.5.8.2, 16.5.9, 16.6 to 16.6.7, 16.6.8.1, 16.6.8.2, 16.6.9, 16.7 to 16.7.7, 16.7.8.1, 16.7.8.2, 16.7.9; Section 19; Section 20; Sections 21.1 to 21.2.1, 21.3; Section 22; Section 23; Sections 24.1 to 24.4, 24.6; Sections 25.2, 25.3; Sections 26.1, 26.3; Section 27.
- Marius Phillips, MAusIMM (CP) (227570), Senior Principal Consultant – Mineral Processing, employed by Stantec Australia Pty Ltd, was responsible for: Sections 1.12, 1.16.2, 1.17.2, 1.18.3, 1.19, 1.19.3, 1.19.4; Section 2; Section 10.7; Section 11.3; Section 13; Section 17; Section 24.5; Section 25.4; Section 26.4, 26.5; Section 27.
- Alwyn Scholtz, B.Eng., MSc, Pr.Eng, ECSA (20150110) employed by DRA Global as Study Manager, was responsible for: Section 1.19; Section 2; Sections 16.3.8.3 to 16.3.8.10, 16.4.8.3 to 16.4.8.10, 16.5.8.3 to 16.5.8.10, 16.6.8.3 to 16.6.8.11, 16.7.8.3 to 16.7.8.9; Sections 18.1 to 18.1.6, 18.1.8 to 18.2.3; Sections 21.1, 21.2.2 to 21.3; Section 25.5; Section 26.6; Section 27.
- Andreas Savvas, BSc Eng (Civil), MSc Eng, M.ASCE, MSAIMM, MIMMM (691401) employed by Epoch Resources as Project Director, was responsible for: Section 1.19; Section 2; Sections 18.1.7, 18.2.4; Section 27.

Signature Page

Project Name:	Kamoa-Kakula Project
Title:	Kamoa-Kakula Integrated Development Plan 2023
Location:	Lualaba Province Democratic Republic of the Congo
Effective Date of Technical Report:	6 March 2023

/s B F Peters

Date of Signing:

Bernard Peters FAusIMM (201743), Technical Director - Mining, OreWin Pty Ltd

/s J Witley

Date of Signing:

Jeremy Witley Pr.Sci.Nat SACNASP, FGSSA, HOD, Mineral Resources, The MSA Group

/s W Joughin

Date of Signing:

William Joughin, FSAIMM (55634), employed by SRK Consulting (South Africa) (Pty) Ltd as Corporate Consultant

/s C Smith

Date of Signing:

Curtis Smith MAusIMM (CP) (311458), Principal Mining Engineer, OreWin Pty Ltd

/s M Phillips

Date of Signing:

Marius Phillips, MAusIMM (CP) (227570), Senior Principal Consultant – Mineral Processing, Stantec Australia Pty Ltd

/s A Scholtz

Date of Signing:

Alwyn Scholtz B.Eng, M.Sc, Pr.Eng (ECSA nr.20150110) Study Manager, DRA Global

/s A Savvas

Date of Signing:

Andreas Savvas, BSc Eng (Civil), MSc Eng, M.ASCE, MSAIMM, MIMMM (691401)

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1.1 Introduction

The Kamoa-Kakula Integrated Development Plan 2023 (Kamoa-Kakula IDP23) 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 Mutshatsha territory, in the 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 and blister copper.

The previous Technical Report on the Project was the Kamoa-Kakula Integrated Development Plan 2020 with an effective date of 13 October 2020.

1.2 Kamoa-Kakula Integrated Development Plan 2023

The Kamoa-Kakula IDP23 provides updates to the Project Mineral Reserves, and the studies at Pre-feasibility (PFS), and Preliminary Economic Assessment (PEA) stages. The following are the key features of the Kamoa-Kakula IDP23:

- Kamoa-Kakula 2023 PFS (Plant expansion to 19.2 Mtpa, smelter, and five producing mines).
- Kamoa-Kakula 2023 PEA (Plant expansion to 19.2 Mtpa, smelter, and nine producing mines).

An overview of deposits included within the Kamoa-Kakula 2023 PFS (outlined by blue dotted line) and Kamoa-Kakula 2023 PEA (outlined by green dotted line) is shown in Figure 1.1.

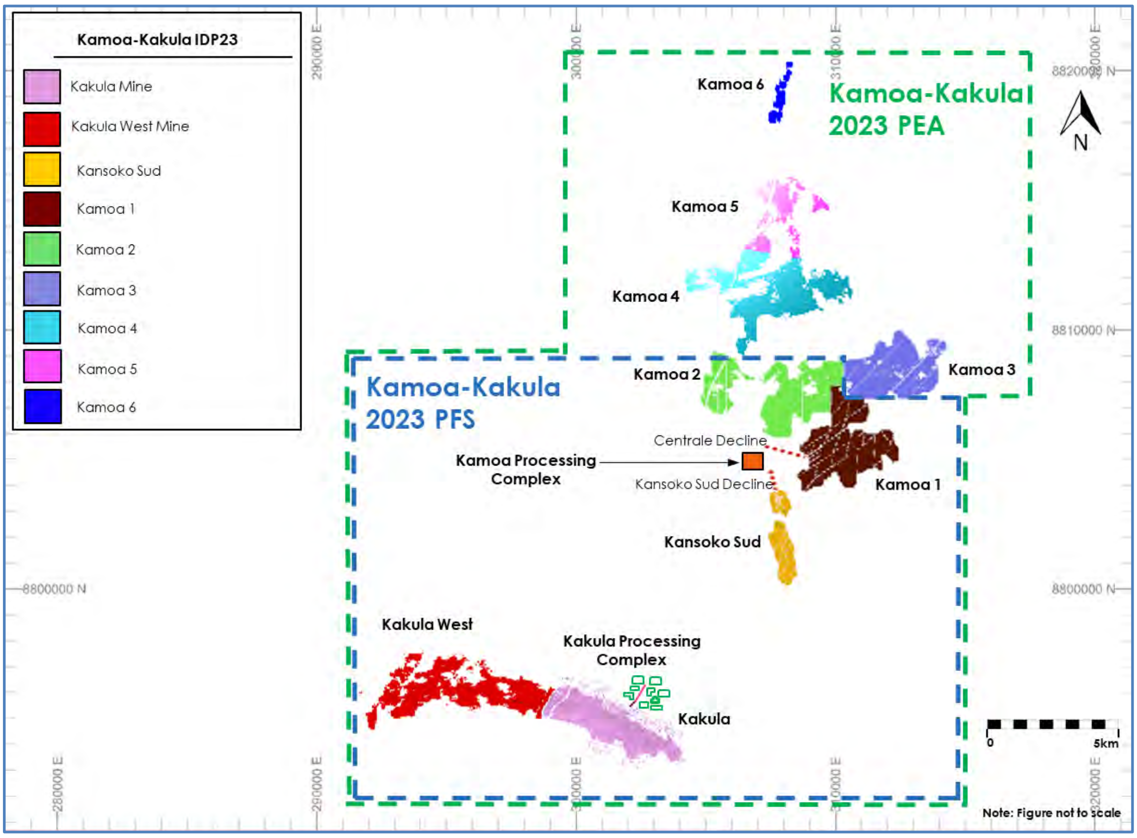
Each of the Kamoa-Kakula deposits are separate underground mines at Pre-feasibility, and Preliminary Economic Assessment level of development. The Kamoa-Kakula 2023 PFS, and Kamoa-Kakula 2023 PEA study include separate capital and operating costs, and assumptions for underground mining, processing plant, smelter and infrastructure.

The Kamoa-Kakula 2023 PFS, and Kamoa-Kakula 2023 PEA, analyses a production case with an expansion of the Kakula/Kamoa concentrator processing facilities, and associated infrastructure to 19.2 Mtpa. Each scenario includes a smelter, and up to nine separate underground mining operations with associated capital and operating costs. The details of the Kamoa-Kakula 2023 PFS, and the Kamoa-Kakula 2023 PEA, are provided in Section 16, and Section 24, respectively.

The nine mines are listed below:

- Kakula Mine (PFS 9.2 Mtpa)
- Kakula West Mine (PFS 6.2 Mtpa)
- Kamoa 1 Mine (PFS 6.0 Mtpa)
- Kansoko Sud Mine (PFS 2.0 Mtpa)
- Kamoa 2 (PFS 6.0 Mtpa)
- Kamoa 3 (PEA 6.0 Mtpa)
- Kamoa 4 (PEA 6.0 Mtpa)
- Kamoa 5 (PEA 3.0 Mtpa)
- Kamoa 6 (PEA 1.0 Mtpa)

Figure 1.1 Kamoa-Kakula IDP23 Mining Locations

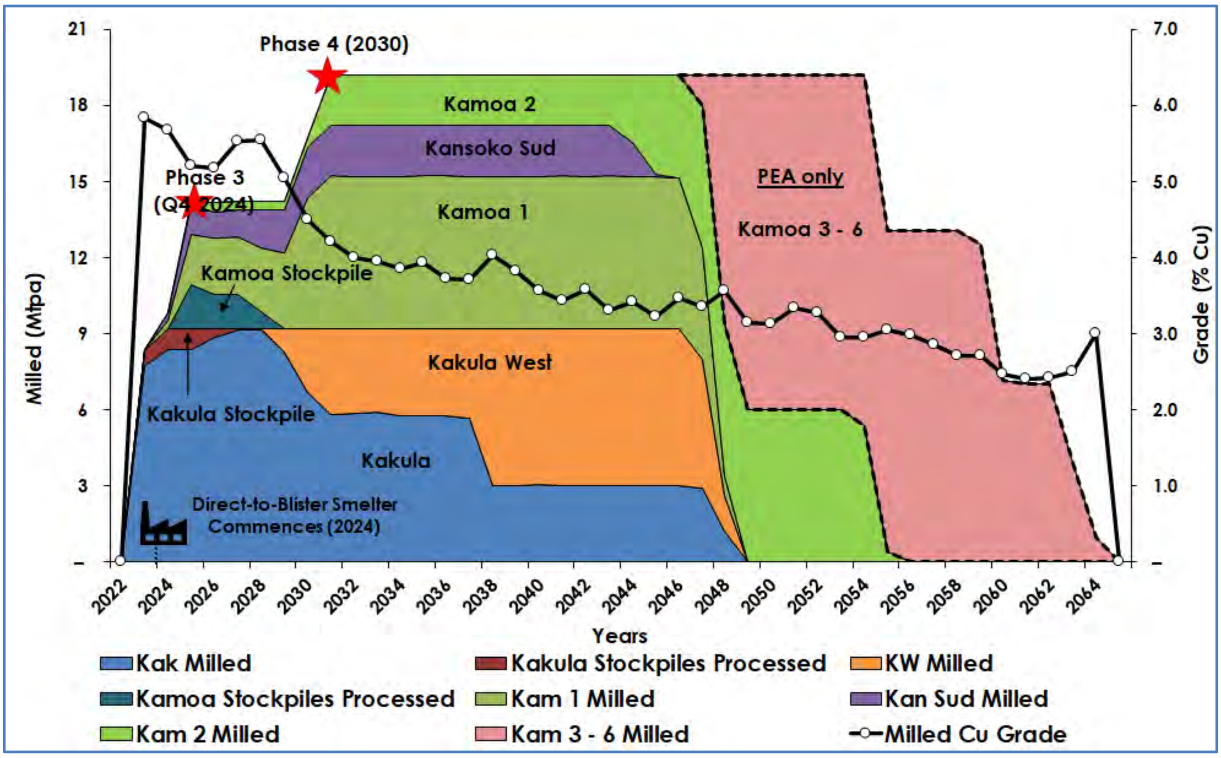


OreWin, 2023.

The Kamoa-Kakula 2023 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.

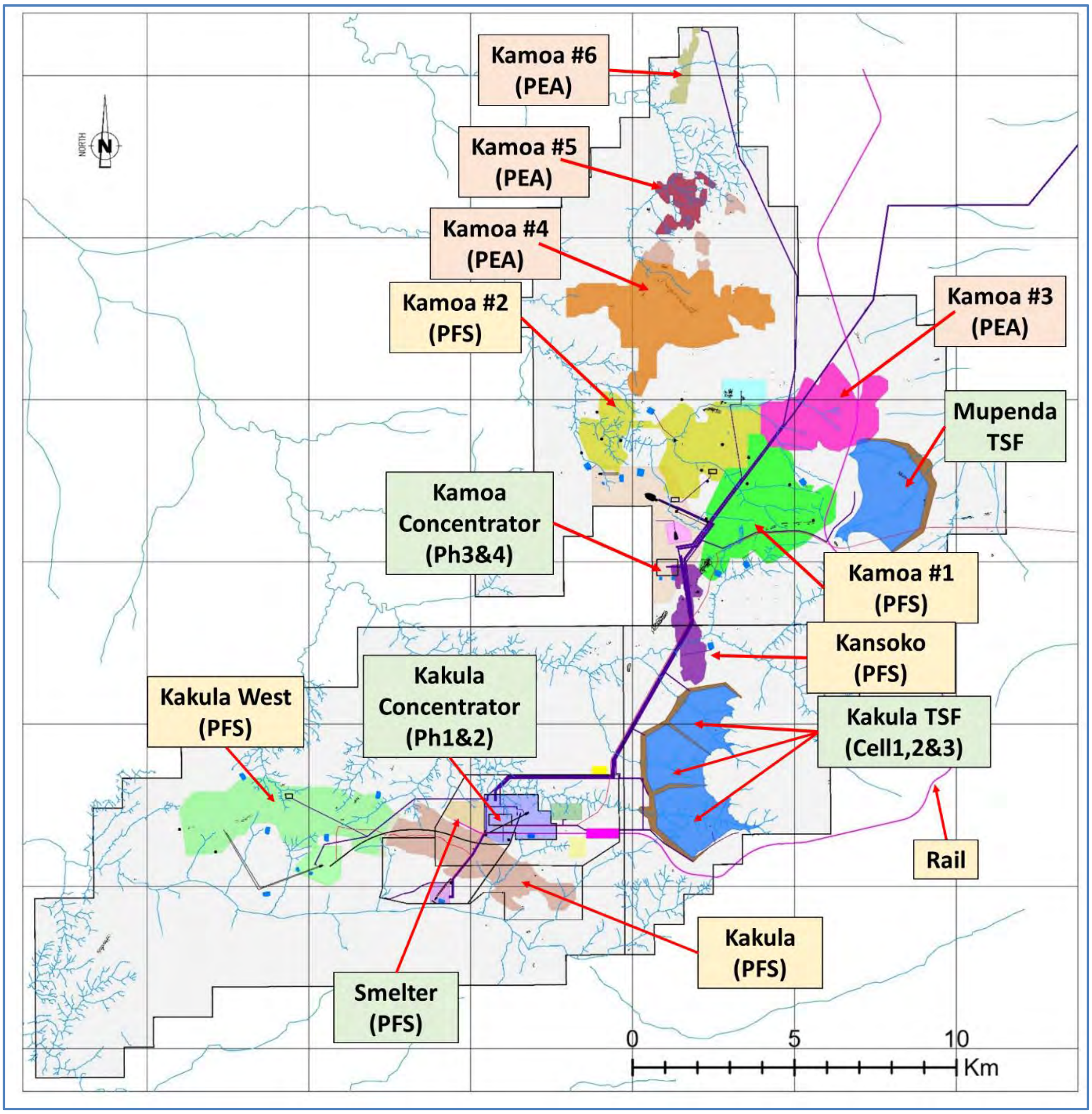
The Kamoa-Kakula IDP23 development scenario is shown in Figure 1.2. A site plan showing the locations of the mines and key infrastructure for the Kamoa-Kakula deposits is shown in Figure 1.3. The Kamoa-Kakula IDP23 production and economic analysis results are shown in Table 1.1.

Figure 1.2 Kamoa-Kakula IDP23 Long-Term Development Scenario



OreWin, 2023.

Figure 1.3 Kamoa-Kakula IDP23 Site Plan



Kamoa Copper SA, 2023.

Table 1.1 Kamoa-Kakula IDP23 Results Summary

Item	Unit	Kamoa-Kakula 2023 PFS	Kamoa-Kakula 2023 PEA
Total Processed			
Quantity Milled	kt	476,195	657,428
Copper Feed Grade	%	3.94	3.70
Total Concentrate Produced			
Copper Concentrate Produced	kt (dry)	37,802	50,761
Copper Recovery	%	86.62	86.45
Copper Concentrate Grade	%	43.05	41.45
Contained Metal in Concentrate	Mlb	35,875	46,384
Contained Metal in Concentrate	kt	16,273	21,040
Annual Average (2023-2024)			
Ore Milled	kt	9,106	9,106
Copper Feed Grade	%	5.75	5.75
Copper Concentrate Produced	kt (dry)	917	917
Contained Copper in Concentrate	Mlb	1,004	1,004
Contained Copper in Concentrate	kt	455	455
C1 Cash Cost	US$/lb. payable	1.45	1.45
EBITDA	US$M	2,015	2,015
Annual Average (2025-2029)			
Ore Milled	kt	14,194	14,194
Copper Feed Grade	%	5.30	5.30
Copper Concentrate Produced	kt (dry)	1,431	1,431
Contained Copper in Concentrate	Mlb	1,442	1,442
Contained Copper in Concentrate	kt	654	654
C1 Cash Cost	US$/lb. payable	1.15	1.15
EBITDA	US$M	3,522	3,522
Annual Average (First 10-Years)			
Ore Milled	kt	14,428	14,428
Copper Feed Grade	%	4.94	4.94
Copper Concentrate Produced	kt (dry)	1,379	1,379
Contained Copper in Concentrate	Mlb	1,368	1,368
Contained Copper in Concentrate	kt	620	620
C1 Cash Cost	US$/lb. payable	1.22	1.22
EBITDA	US$M	3,151	3,151
Key Financial Results			
Remaining Phase 3 Capital Costs	US$M	3,037	3,037
Phase 4 Capital Costs Capital Costs	US$M	1,553	1,553
Sustaining Capital Costs	US$M	5,583	8,858
LOM Avg. C1 Cash Cost	US$/lb Payable Cu	1.31	1.32
LOM Avg. Total Cash Costs	US$/lb Payable Cu	1.52	1.53
LOM Avg. Site Operating Costs	US$/t Milled	72.75	70.57
After-Tax NPV8%	US$M	19,062	20,224
Project Life	Years	33	42

1.3 Project Development

Following the successful completion of Phase 1 and Phase 2 concentrators, the plant processing capacity was increased to 7.6 Mtpa. A debottlenecking project was completed on the Kakula Plant in early 2023, and has increased the processing capacity to 9.2 Mtpa.

Current construction work includes the remaining Phase 2 infrastructure non-critical work (7.6 Mtpa throughput already achieved), Kakula Debottlenecking and the start of Phase 3. Remaining Phase 2 will be concluded at the end of 2023 with the following work ongoing:

- Mining infrastructure (Kakula conveyors, Kakula dewatering, Kakula primary ventilation Kansoko Sud dewatering and Kansoko Sud primary ventilation).
- Surface infrastructure (Kansoko Sud water systems and electrical, Kakula power generation, Kakula electrical infrastructure, Kakula stormwater drains and Kakula truck park).
- TSF Raise 3 and 4.

Phase 3 basic engineering was completed in 2022, and engineering and construction work has started on the smelter, concentrator, surface infrastructure and mining. Current progress on these is as follows:

Concentrator:

- Earthworks for the Phase 3 concentrator plant are well advanced, and orders have been placed on all the long lead equipment. Civil and structural designs are nearing completion and civil construction started on the ROM Stockpile, secondary screening and HPGR stockpile.

Smelter:

- All terracing earthworks for the smelter complex were completed in 2022, and the civil construction is now well advanced with all piling complete, and foundations for the DBF copper flash smelting furnace, and downstream electric slag cleaning furnace, nearing completion. The erection of structural steel and the DBF furnace is due to start in March 2023. The first batch of DBF furnace steel arrived on-site in January 2023. All major equipment has been ordered and is now being manufactured, while construction is on schedule to commission the smelter by the end of 2024.

General Surface infrastructure:

- Other surface infrastructure to support Phase 3 operations include a dedicated 220 kV substation at Kamoa, a new backfill plant, expansion of the existing camps (at Kakula and Kamoa), and a new camp adjacent to the smelter (total of over 2,500 beds), and expansion of the existing Kakula tailings storage facility (TSF).

Power (Inga Hydropower):

- Rehabilitation work is ongoing at turbine (#5) of the existing Inga II hydropower facility on the Congo River, which will generate an additional 178 megawatts (MW) of renewable hydropower, which underpins the Phase 3 power requirement, including the smelter. The refurbishment is scheduled for completion in Q4'24, to align with the commissioning of the Phase 3 concentrator and smelter.

Mining and Mining infrastructure:

- Mining infrastructure at the Kamoa 1 decline started with construction of the box-cut dewatering system, shotcrete plant, emulsion plant, 150 m storm water dam and a surface fuel bay.
- Ongoing construction at Kakula includes the extension of the East conveyor system and dewatering infrastructure.

1.4 Property Description and Location

The Project is situated in the Mutshatsha territory in the 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 approximately 397.4 km^2. Title of the exploitation licences is held by Kamoa Copper. The Exploitation Licences were approved on 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 periods not exceeding 15-years each, until the end of the mine's life.

Kamoa Copper is a subsidiary of Kamoa Holding Limited (80%) and the DRC (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 DRC. This transfer was pursuant to the 2002 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. This increased the DRC 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 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 exploitation permits are subject to taxes, customs and levies defined by the mining code and mining regulations, for all mining activities carried out by the holder in the DRC.

The exploitation permits were originally granted in 2012 under Law No.007/2002 dated 11 July 2002 on the 2002 Mining Code.

On 9 March 2018, Law No. 18/001 amending the 2002 DRC Mining Code, was promulgated (the 2018 DRC Mining Code).

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 (Ng1.1.1.1), sandstone and siltstone (Ng1.1.1.2), and clast-poor diamictite (Ng1.1.1.3). The lowermost clast rich diamictite (Ng1.1.1.1) unit generally hosts lower grade (<0.5% TCu) mineralisation. Most of the higher grade mineralisation occurs within the clast-poor diamictite (Ng1.1.1.3) unit, or in the sandstone and siltstone (Ng1.1.1.2) interbeds that are locally present between the clast rich (Ng1.1.1.1) and clast-poor (Ng1.1.1.3) 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 (Cu2S), bornite (Cu5FeS4), and chalcopyrite (CuFeS2), 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 programme 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 surveys 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 MSA Qualified Person (the MSA 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 20 July 2022, with the assay database closed on 13 December 2022, and the model was completed as of 14 December 2022.

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

As at 2 December 2022, there were 2,808 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 645 drillhole intercepts (one intercept per drillhole).

The 1,165 holes not included in either the January 2020 Kamoa, or the December 2022 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), 139 drillholes have been completed inside of the modelled area at Kamoa. An additional 42 drillholes at Kakula have been completed after the closure of the database on 20 July 2022, primarily as infill drilling in close proximity to the current underground development, or as infill in planned future mining areas at Kakula West.

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%. Collar positions for all completed holes are surveyed by an independent professional surveyor SD Geomatique using a differential GPS which is accurate to within 20 mm. Only three drillholes used in the Mineral Resource estimate lacked an independently surveyed collar position.

The Kakula drillhole collars have been surveyed by SD Geomatique and E.M.K. Construction SARL. As of 20 July 2022, three drillholes used in the Mineral Resource estimate lacked an independently surveyed collar position. Visual inspection of the Kakula core by the MSA QP documented the core recovery to be excellent.

In the opinion of the MSA QP, the quantity and quality of the lithological collar, and down-hole 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.

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 MSA QP, the sampling and analytical methods are acceptable, are consistent with industry standard practices, and are adequate for Mineral Resource estimation.

1.11 Data Verification

MSA reviewed the sample chain of custody, quality assurance and control (QA/QC) procedures, and qualifications of analytical laboratories. MSA is of the opinion that the procedures and QA/QC control are acceptable to support Mineral Resource estimation. MSA also audited the assay database, core logging, and geological interpretations during the site visit conducted by the MSA QP, and has found no material issues with the data as a result of these audits.

In the opinion of the MSA 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 resource has a long history of metallurgical testwork undertaken by various parties, which focussed on the metallurgical characterisation and flow sheet development, for the processing of hypogene and supergene copper ores. These investigations culminated in the development of the IFS4a flow sheet in support of the Kamoa PFS (March, 2016). During 2016, Kamoa Copper SA discovered the Kakula deposit, which has significantly higher copper head grades, compared to the Kamoa deposit. Consequently, the Kakula project was fast-tracked. Metallurgical testwork on the Kakula deposit was initiated in 2016. The Kakula Phase 1 and Phase 2 concentrators have been successfully commissioned in 2021/2022, and since ramped up to design throughput.

1.12.1 Kakula

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 pre-feasibility 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 metallurgical drilled core 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 Kamoa-Kakula 2023 PFS recovery estimate for Kakula material is based on the historical Kakula 2019 PFS flow sheet development testwork, historical Kakula variability campaign, as well as current operating data. The recovery model is based on a final product grade of 50.0% Cu. The average Cu recovery over the life-of-mine to produce a 50.0% Cu concentrate was determined as 86.3% from a 4.42% Cu head grade.

1.12.2 Kamoa

Between 2010 and 2015 a series of metallurgical testwork programmes, defined as Phases 1 to 5, were completed on Kamoa drill core sample and focussed on metallurgical characterisation and flow sheet development, for processing the hypogene and supergene material. During this period the 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 Expert Process Solutions (XPS) Laboratories. The Phase 6 samples best represent ores to be processed, according to the early years of the Kamoa 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.

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. The reduction in recovery was attributed to aging of the sample, as was evident during further testwork to evaluate the effect of mine water on the flotation response.

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 to both ore bodies.

The Kamoa-Kakula 2023 PFS includes for the treatment of Kamoa material in the originally developed IFS4a flow sheet. The recovery estimate for Kamoa material is based on the historical flow sheet development work. The average Cu recovery over the life-of-mine to produce a 37.0% Cu concentrate was determined as 87.0% from a 3.49% Cu head grade.

1.13 Mineral Resource Estimates

The Kamoa and Kakula resource models are based on the same 3D estimation methodology. The locations 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 seven 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/ threshold 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 (Indicated Mineral Resource), covers an area of 55.2 km^2 and the Inferred Mineral Resource covers an area of 21.8 km^2. The deposit remains open laterally.

At a 1% copper cut-off, the Kakula Measured Mineral Resource covers an area of 2.2 km^2, the Indicated Mineral Resource covers an area of 21.7 km^2, and the Inferred Mineral Resource covers an area of 5.5 km^2. The deposit remains open laterally, and the southern parts of the Kamoa-Kakula exploitation licence area are virtually untested.

For reporting Mineral Resources, MSA used a 1% TCu cut-off grade as a base case. At mines on the Zambian Copperbelt such as Konkola, Nchanga, Nkana, and Mufulira, a 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. MSA further tested the cut-off grade using reasonably assumed cost and revenue assumptions based on PEA inputs.

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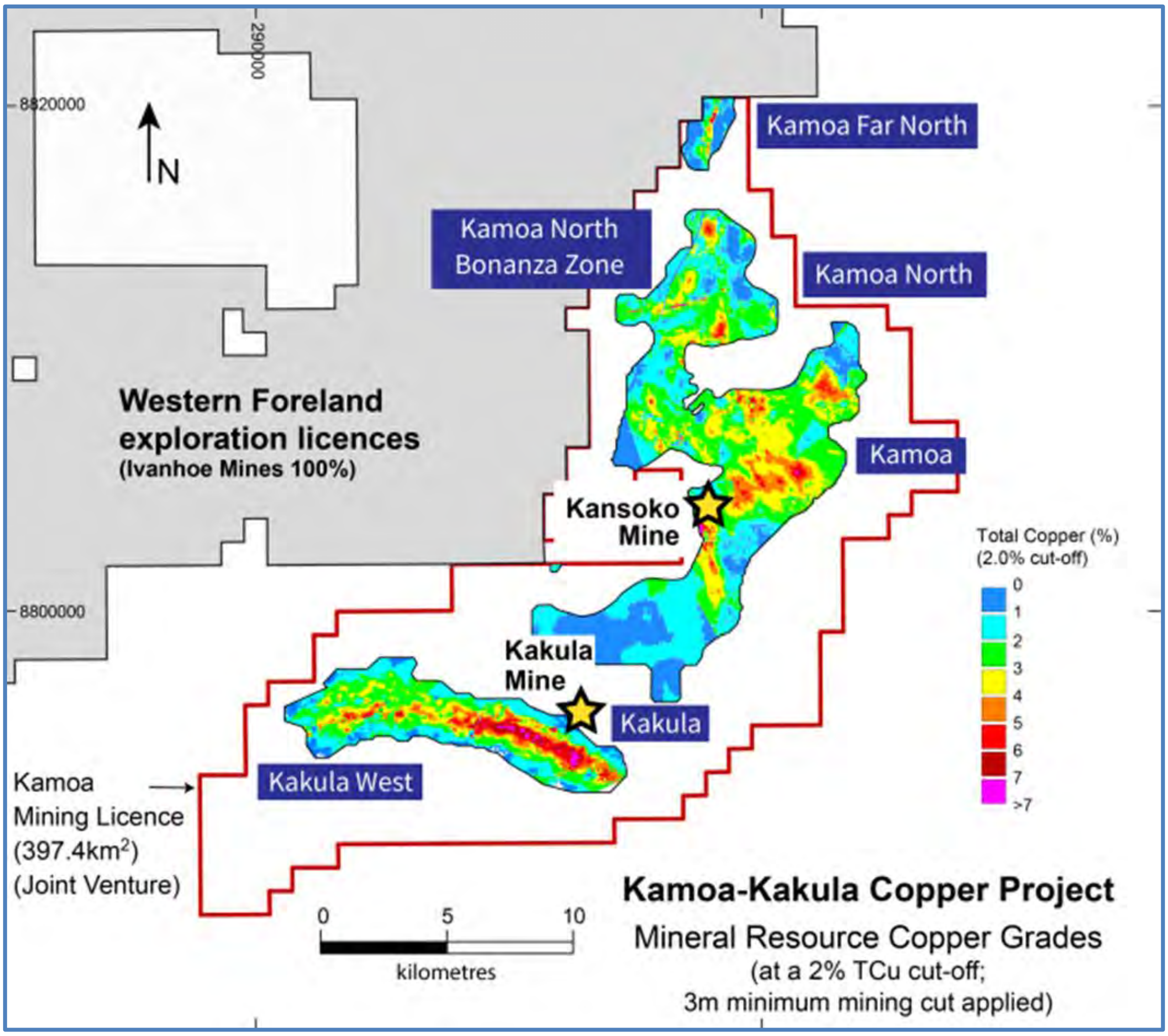
Figure 1.4 Kamoa-Kakula Exploitation Licence, Showing the Kamoa, Kakula, Kakula West, Kamoa North Bonanza Zone and Kamoa Far North Mineral Resource Areas.



Ivanhoe, 2023.

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 14 December 2022. On 31 December 2022, the model was depleted to account for annual production, and the Mineral Resource has an effective date of 31 December 2022.

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 1.2 Kamoa and Combined Kakula Indicated and Inferred Mineral Resources

Deposit	Category	Tonnage (Mt)	Area (km^2)	Copper (%)	Contained Copper (kt)	Contained Copper (billion lbs)
Kamoa	Measured	-	-	-	-	-
	Indicated	760	55.2	2.73	20,800	45.8
	Inferred	235	21.8	1.70	4,010	8.8
Kakula	Measured	90	2.2	3.13	2,810	6.2
	Indicated	540	21.7	2.65	14,300	31.6
	Inferred	75	5.5	1.60	1,200	2.6
Total Kamoa-Kakula Project	Measured	90	2.2	3.13	2,810	6.2
	Indicated	1,300	76.9	2.70	35,100	77.4
	Inferred	310	27.3	1.68	5,210	11.5

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 that were reviewed by Jeremy Witley, Pr.Sci.Nat SACNASP, FGSSA, who is the Qualified Person for the Mineral Resource estimate. The cut-off date for drill data for the Kamoa estimate is 20 January 2020. The cut-off date for the drill data for the Kakula estimate is 20 July 2022, with the assay table updated as of 13 December 2022. On 31 December 2022, the Mineral Resource was depleted to account for annual production; the Mineral Resource has an effective date of 31 December 2022. 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 the following assumptions: copper price $4.00/lb; employment of underground mechanised drift-and-fill mining methods; copper blister and concentrates will be produced and sold; average metallurgical recovery is 87.5%; mining costs are assumed to be $38/t; concentrator, tailings treatment, and general and administrative costs are assumed to be $15/t; smelter, refining and transport costs are assumed to be $13.5/t of ore at the cut-off grade; royalty of 3.5%, export tax of 1% and concentrate tax of $100/t NSR /t concentrate.
3. Mineral Resources are reported for Kakula 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 the following assumptions: copper price $4.00/lb; employment of underground mechanised drift-and-fill mining methods, and that copper blister and concentrates will be produced and sold; average metallurgical recovery is 85.5%; mining costs are assumed to be $38/t; concentrator, tailings treatment, and general and administrative costs are assumed to be $15/t; smelter, refining and transport costs are assumed to be $9.5/t of ore at the cut-off grade; royalty of 3.5%, export tax of 1% and concentrate tax of $100/t NSR /t concentrate.
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 mass percentages.
6. Approximate drillhole spacings are 800 m for Inferred Mineral Resources, 400 m for Indicated Mineral Resources, and 100 m or underground exposure for Measured Mineral Resources.
7. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.

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 or rift geometries on lithology and grade continuity assumptions. Local faulting could disrupt the productivity of a highly mechanised operation. Ivanhoe plans to study these risks with the underground development in progress at Kamoa and Kakula.
  - Delineation drill programmes 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.
  - In the Kakula south developments, minor offsets across growth faults have been encountered, but adjustments to the mining methods have 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.
- 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, or across localised folds.
- Metallurgical recovery assumptions at Kamoa:
  - Variability testwork has been conducted on portions of the Kamoa deposit, and therefore the average recoveries used in the cut-off grade for assessment may differ from actual performance. Areas of supergene mineralisation are likely to require different metallurgical parameters, however these areas make up only a small part of the deposit.
- Metallurgical recovery assumptions at Kakula:
  - Operational performance has confirmed study testwork results regarding recoveries and concentrate grades.
  - There is no supergene mineralisation currently identified at Kakula that requires a dedicated recovery model separate from the hypogene recovery prediction method.
  - Further testwork on the Kamoa material indicated that the Kansoko and Kakula material can be processed in a common concentrator circuit.
- 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 south-eastern boundary of the deposit within the Mineral Resources is defined solely by the current limit of drilling. Large portions of the licence area south of Kakula remain untested and there is potential for discovery of additional mineralisation.

1.15 Mineral Reserves

1.15.1 Kamoa-Kakula Project Mineral Reserve

A Mineral Reserve has been created for each deposit within the Kamoa-Kakula 2023 PFS, which consists of Kakula, Kakula West, Kamoa 1, Kansoko Sud, and Kamoa 2. The mine design and scheduling focussed on maximising the grade profile at full production rate, with emphasis on further maximising grade early in the mine life. The tonnes and grades were calculated for the mining shapes, 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 1.3.

Only the Indicated portion of the Mineral Resources 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 31 December 2022. The Kamoa 2022 Mineral Resource estimate with an effective date 31 December 2022 was not completed in time to allow it to be included in the Kamoa-Kakula 2023 PFS.

The increase in Mineral Reserves since the 2020 Mineral Reserve in the Kamoa-Kakula Integrated Development Plan 2020 can be attributed to an increased height (7.5 m) of the second lift at Kakula, the redefining of mine boundaries at Kamoa, and the addition of Kakula West and Kamoa 2.

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 costs 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.

Table 1.3 Kamoa-Kakula 2023 PFS Mineral Reserve

Classification	Ore (Mt)	Copper (%)	Copper (Contained Mlb)	Copper (Contained kt)
Proven Mineral Reserve	–	–	–	–
Probable Mineral Reserve	472	3.94	41,055	18,622
- Kakula	138	4.79	14,580	6,613
- Kakula West	90	3.87	7,647	3,469
- Kamoa 1	121	3.74	9,963	4,519
- Kansoko Sud	38	3.70	3,088	1,401
- Kamoa 2	86	3.05	5,778	2,621
Probable Mineral Reserve	472	3.94	41,055	18,622

1. The effective date of the Mineral Reserve statement is 31 December 2022.
2. The long-term copper price used for calculating the financial analysis is $3.70/lb. The analysis has been calculated with assumptions for an on-site smelter and excess concentrate sold to external smelters. Realisation costs include refining and treatment charges, deductions and payment terms, blister and concentrate transport, metallurgical recoveries, and royalties.
3. For mine planning, the copper price used to calculate block model Net Smelter Return (NSR) is $3.10/lb.
4. An elevated cut-off of $100.00/t NSR was used to define the stoping blocks. A marginal cut-off of $80.00/t NSR was used for development.
5. Indicated Mineral Resources were used to report Probable Mineral Reserves from the January 2020 Mineral Resource.
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 may result in apparent differences between tonnes, grade, and contained metal content.

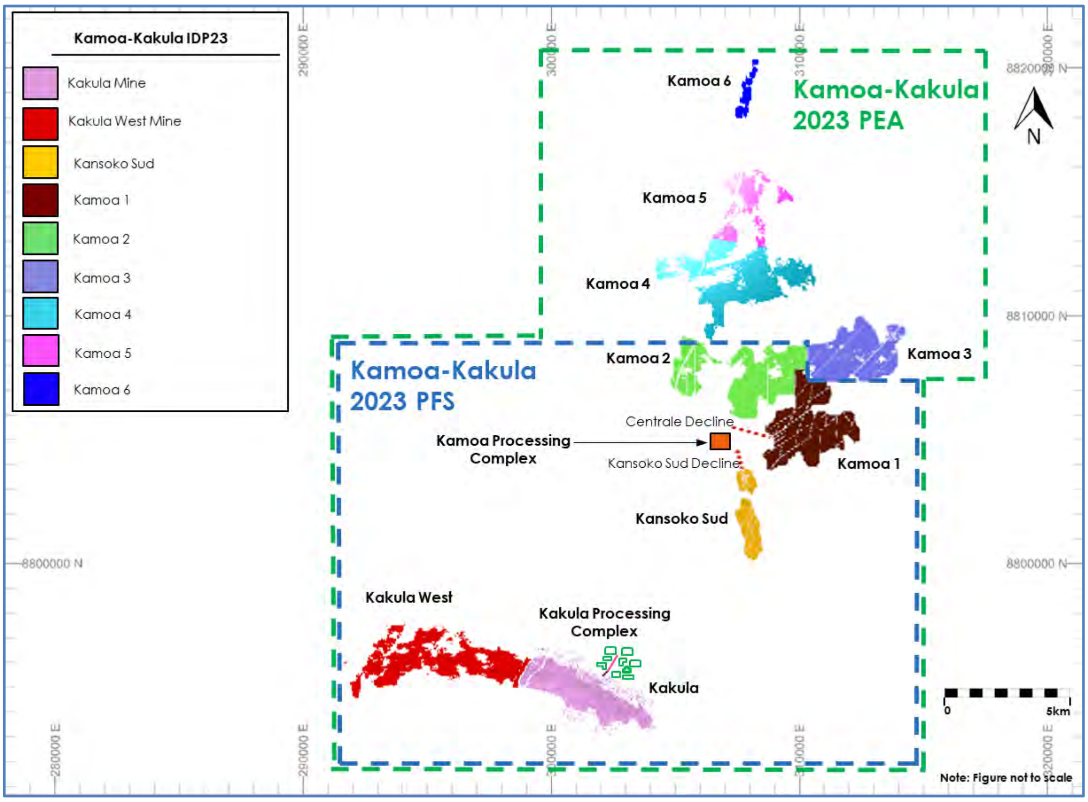
1.16 Kamoa-Kakula 2023 PFS

The Kamoa-Kakula 2023 PFS analyses a production case with an expansion of the Kamoa-Kakula concentrator processing facilities, and associated infrastructure to 19.2 Mtpa, and includes a smelter, and five separate underground mining operations with associated capital and operating costs. Overview of deposits included within the Kamoa-Kakula 2023 PFS (outlined by blue dotted line) is shown in Figure 1.5.

The five mines are listed below:

- Kakula Mine (PFS 9.2 Mtpa).
- Kakula West Mine (PFS 6.2 Mtpa).
- Kamoa 1 Mine (PFS 6.0 Mtpa).
- Kansoko Sud Mine (PFS 2.0 Mtpa).
- Kamoa 2 (PFS 6.0 Mtpa).

Figure 1.5 Kamoa-Kakula 2023 PEA / PFS Mining Locations



OreWin, 2023.

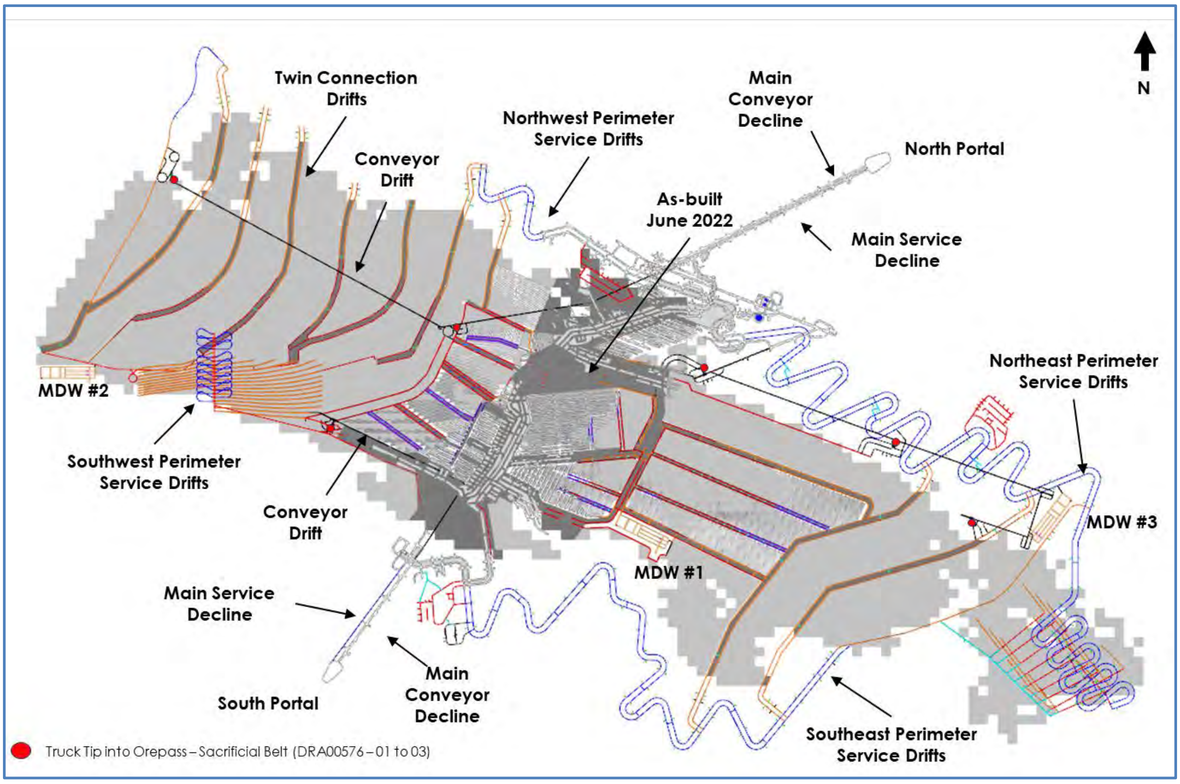
1.16.1 Kamoa-Kakula 2023 PFS Mining

The Kamoa-Kakula 2023 PFS assesses mining multiple deposits on the Kamoa-Kakula Project as an integrated, 19.2 Mtpa mining, processing, and smelting complex. Kakula Mine has been expanded to 9.2 Mtpa to meet the Phase 1 and 2 concentrators requirements (9.2 Mtpa), Kakula West will come begin production in 2029 to maintain Kakula concentrator capacity at 9.2 Mtpa. Kansoko Sud Mine, which is current being developed, and Kamoa 1, and Kamoa 2 mines, will feed the Phase 3 Concentrator, planned for Q4'24, at a capacity of 5.0 Mtpa. The Phase 4 Concentrator also planned at 5.0 Mtpa will be fed by Kamoa 3, Kamoa 4, Kamoa 5, and Kamoa 6 mines. This would bring the Kamoa mining complex capacity to 10 Mtpa together with the Kakula Mining Complex at 9.2 Mtpa to have a combined output of 19.2 Mtpa for the Kakula, and Kamoa mining complex.

The primary mining method for the Kamoa-Kakula deposits (drift-and-fill) was selected based on the results of a trade-off study which compared six different variations of drift-and-fill mining to identify the most suitable. The drift-and-fill method will be utilised as the primary mining method for all Kamoa-Kakula deposits. Identified mining areas with a dip greater than 25° will be mined using the hanging wall access drift-and-fill (HWAD&F) method. To establish the drift-and-fill mining method, a pair of twin perimeter declines are driven at a defined offset to the extremities of each deposit. Twin connection drifts are then developed across the target orebody. The selected drift-and-fill mining methods are explained in detail in Section 16.2.

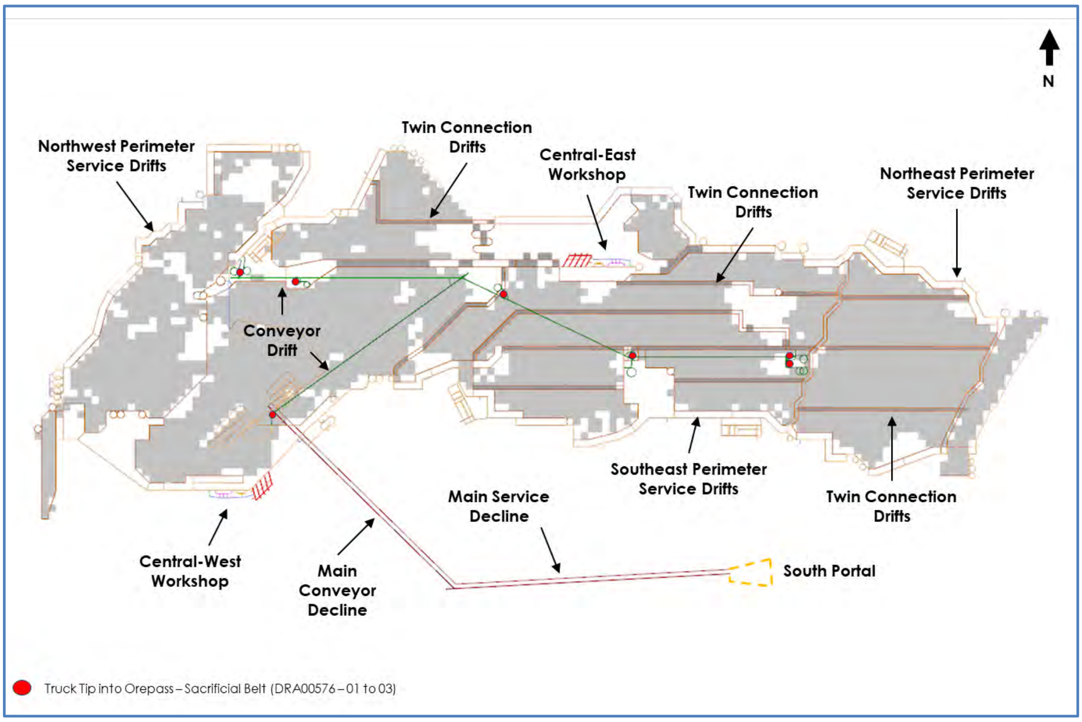
Figure 1.6, Figure 1.7, Figure 1.8, Figure 1.9, and Figure 1.10, show the required mine layout to enable the drift-and-fill mining method for each deposit.

Figure 1.6 Kakula Mine Framework



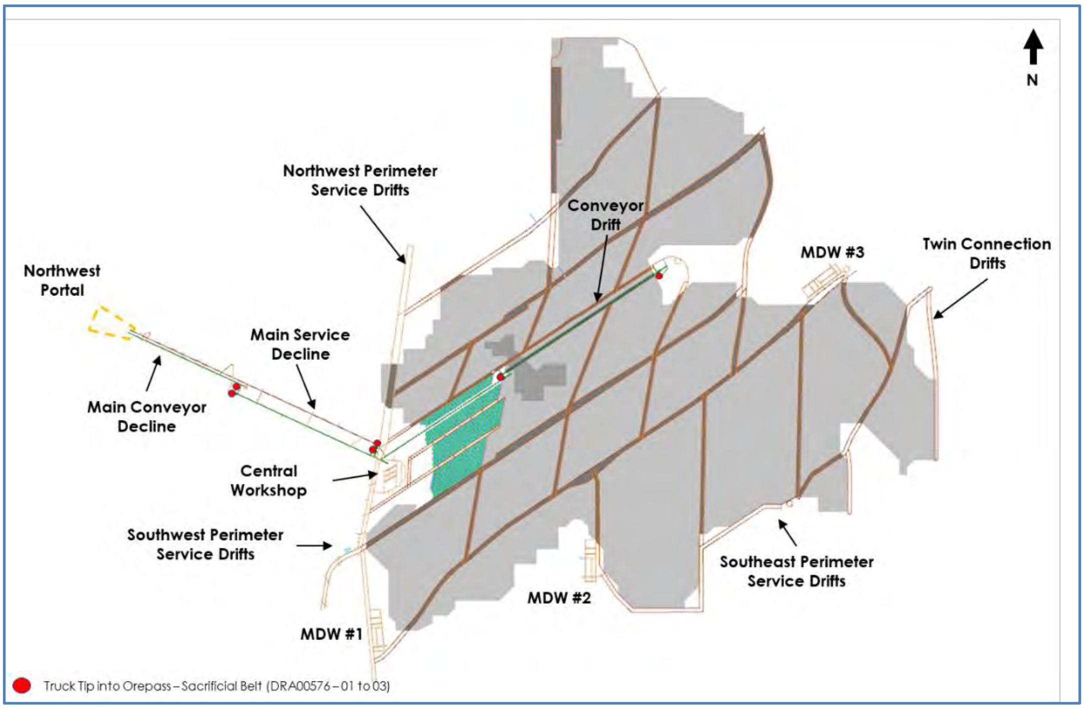
OreWin, 2023.

Figure 1.7 Kakula West Mine Framework



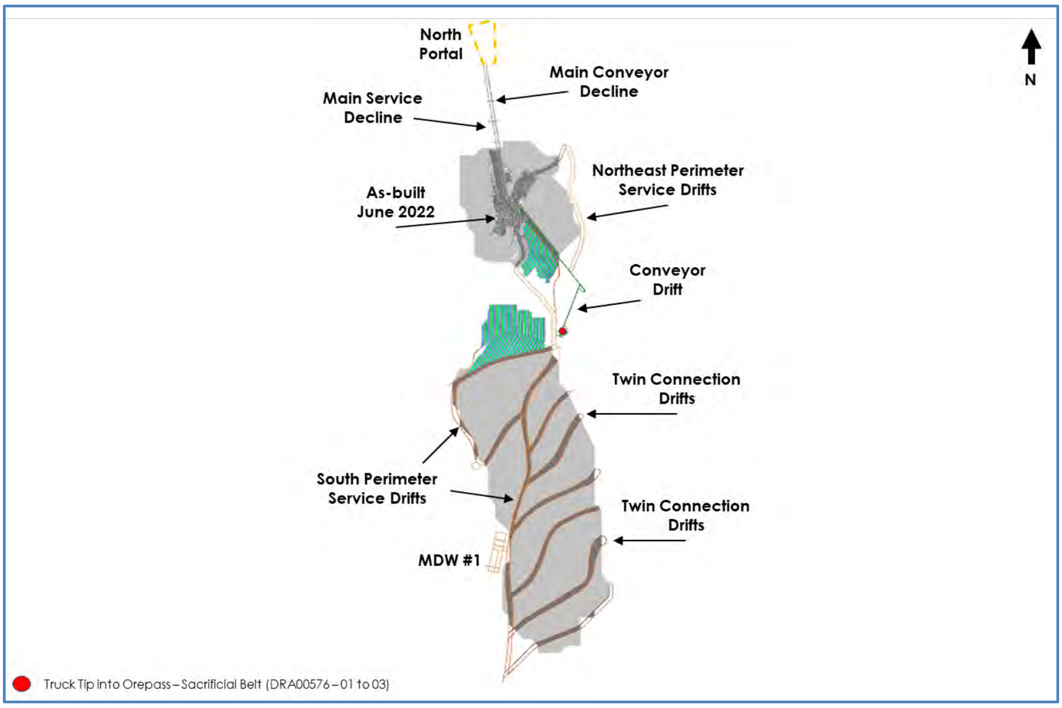
OreWin, 2023.

Figure 1.8 Kamoa 1 Mine Framework



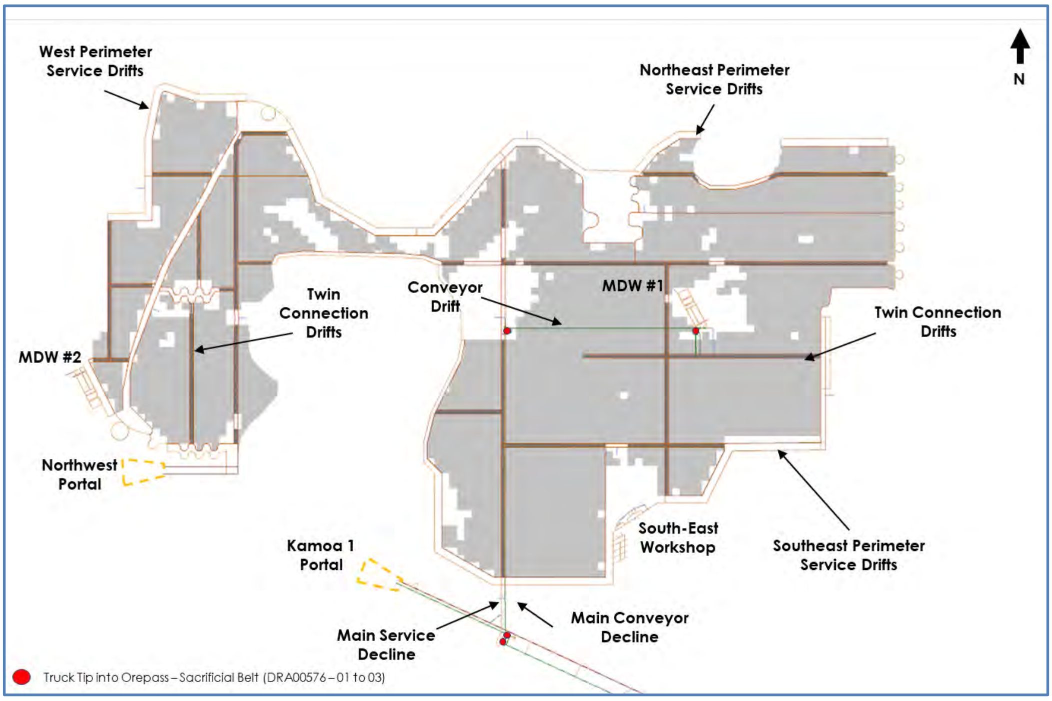
OreWin, 2023.

Figure 1.9 Kansoko Sud Mine Framework



OreWin, 2023.

Figure 1.10 Kamoa 2 Mine Framework



OreWin, 2023.

A probable combined Mineral Reserve of approximately 472.28 Mt grading at 3.94% Cu has been defined in the Kamoa-Kakula mines. Table 1.4 summarises the LOM development and production results for each mine in the Kamoa-Kakula 2023 PFS. The output from the existing Kakula Mine is supported by a detailed rolling mine plan. Indicated production outputs from Kakula West, Kansoko Sud, Kamoa 1, and Kamoa 2 require detailed studies and planning at the execution level to be realised, and although a PFS is a preliminary feasibility study the Kamoa-Kakula 2023 PFS more than adequately meets PFS definitions the NI 43-101 Standards of Disclosure for Mineral Projects.

Table 1.4 Kamoa-Kakula 2023 PFS Development and Production Summary

Kamoa-Kakula 2023 PFS Mining							
Description	Units	Total	Kakula	Kakula West	Kamoa 1	Kansoko Sud	Kamoa 2
Peak Production Rate	Mtpa	-	9.20	6.20	6.00	2.00	6.00
Waste Development							
Waste Development - Lateral	m	339,406	74,526	119,091	44,752	12,442	88,593
Waste Development - Vertical	m	39,216	16,766	11,496	6,042	1,632	3,281
Total Waste Development	m	378,622	91,292	130,587	50,795	14,074	91,874
Total Waste Development	kt	35,150	7,527	12,747	4,893	1,366	8,617
Ore Production							
Ore Development Metres	m	123,554	53,598	9,011	45,388	11,917	3,639
Ore Development	kt	13,540	6,699	854	4,494	1,154	339
Drift-and-Fill Production	kt	444,517	117,282	88,727	116,349	36,658	85,500
HWAD&F Production	kt	14,227	14,227	–	–	–	–
Total Ore Production	kt	472,284	138,207	89,582	120,843	37,813	85,839
Cu Grade	%	3.94	4.79	3.87	3.74	3.70	3.05
S Grade	%	1.80	1.34	1.27	2.48	2.65	1.75
As Grade	%	0.0005	0.0001	0.0001	0.0005	0.0014	0.0011
Fe Grade	%	5.32	4.86	4.88	5.83	6.19	5.43
AsCu Grade	%	0.31	–	–	0.35	0.23	0.28

The following conditions were used in developing the LOM schedules:

- Proximity to the main accesses and initial 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. Each mine contained underground infrastructure consisting of various components such as ore and waste handling systems, dewatering, maintenance shops, fuelling, ventilation, concrete and shotcrete facilities, refuge stations, etc.

1.16.2 Kamoa-Kakula 2023 PFS Recovery Methods

1.16.2.1 Introduction

The Kamoa-Kakula 2023 PFS considers an increase in production capacity from 7.6 Mtpa up to a total of 19.2 Mtpa achieved by debottlenecking of Kakula Phase 1 and Phase 2 to 9.2 Mtpa, and the phased addition of a further 10.0 Mtpa processing facility located at Kamoa.

The processing capacity of each module of the existing Kakula concentrator will be increased from 3.8 Mtpa to 4.6 Mtpa, through several modifications and hydraulic upgrades, to increase the Kakula complex processing capacity to 9.2 Mtpa.

The new Kamoa concentrator will be constructed in a phased approach with two 5.0 Mtpa modules as the mining operations ramp-up to full production of 10.0 Mtpa.

1.16.2.2 Kakula Concentrator

The flow sheet for the Kakula concentrator is shown in Figure 1.11. 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, 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 tailing's storage facility. Backfill utilises on average 40% of tailings and the remaining 60% will be pumped to the tailing's storage facility.

Based on extensive testwork and current operation data, the concentrator is expected to achieve an overall recovery of 86.3%, producing a concentrate grading 50.0% copper. Kakula concentrate also benefits from having very low deleterious elements, including arsenic levels of 0.02%.

1.16.2.3 Kamoa Concentrator

The Kamoa concentrator design is similar to the existing Kakula concentrator with the difference in the position of the regrind circuit, as per Figure 1.12.

The Kamoa circuit incorporates a run-of-mine stockpile, followed by primary cone crushers operating in closed circuit with vibrating screens to produce 100% passing 50 millimetres (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 scavenger concentrate together with the tailings from the high-grade cleaner stage are combined and re-ground to a P_{80} of 10 µm prior to further upgrading in the scavenger cleaner circuit. The regrind circuit product is combined with the scavenger recleaner tailings as feed to the scavenger cleaner circuit. Provision is made for the scavenger cleaner circuit to produce a medium-grade concentrate or low-grade concentrate, depending on the feed grade to the circuit. The medium-grade concentrate reports to the final concentrate, while the low-grade concentrate produced from the scavenger cleaner circuit reports for final cleaning in the low entrainment scavenger recleaner stage. The scavenger recleaner concentrate is then combined with the high-grade cleaner concentrate and medium-grade scavenger cleaner concentrate to form final concentrate. The final concentrate is then thickened and pumped to the concentrate filters. Final filtered concentrate is trucked to the Kakula smelter complex. The scavenger tailings and scavenger cleaner tailings are combined and thickened prior to being pumped to the backfill plant and/or to the tailing's storage facility. Backfill will utilise approximately half of the tailings, with the remaining amount pumped to the tailing's storage facility.

Based on testwork, the Kamoa concentrator is expected to achieve an overall recovery of 87%, producing a concentrate grading 37% copper. Kamoa concentrate also benefits from having very low deleterious elements, including arsenic levels of 0.03%.

1.16.2.4 Smelter

The Direct-to-Blister Furnace (DBF) technology selected for the Kamoa smelter has greater oxygen and fuel consumption efficiency compared to other technologies, which helps to reduce the overall operating costs of the smelter. It requires fewer major equipment items than some competing processes, enabling it to offer competitive capital costs. Because of the highly oxidative environment in the reaction shaft of the DBF furnace, the resulting slag is relatively high in copper necessitating subsequent treatment in a slag cleaning furnace followed by a slag-mill-flotation circuit.

The overall smelting process proposed for Kamoa-Kakula includes the following main steps:

- Concentrate Handling:
  - Concentrate filter cake storage and blending.
  - Concentrate drying in a steam dryer with bag filter (baghouse) and dedicated exhaust stack.
- Flash Smelting and Slag Cleaning:
  - Feed systems for concentrate, oxygen, diesel, coal, burnt lime and flue dust.
  - Direct to Blister Furnace (DBF) with concentrate burner.
  - DBF waste heat boiler and electrostatic precipitator.
  - Fugitive gas collection and bag filter.
  - Electric Slag Cleaning Furnace with coke feed system, evaporative cooler and dust cyclones for furnace offgas.
  - Blister copper tapping from both DBF and SCF, and transfer to anode furnace.
- Anode Refining Furnaces:
  - DBF and SCF molten blister copper transferred to anode furnaces.
  - Rotary anode furnaces with air-cooled gas coolers.
  - Anode copper to dual casting wheels, containing up to 99.7% copper.
  - Anode Furnace slag recycled to Slag Cleaning Furnace.
- Sulfuric Acid Plant:
  - Double contact, double absorption process for maximum sulfur capture.
- Flue Gas Desulfurisation:
  - All smelter gas streams treated in a de-sulfurisation plant before discharge to ensure minimum SO_2 discharge well within IFC limits).
  - Including the following gas streams: acid plant tail gas, SCF gas, fugitive extraction system and anode refining furnace gas.
  - Desulfurisation of combined gas stream with lime scrubber, producing gypsum for disposal.
  - Wet electrostatic mist precipitator before discharging gas to the main smelter exhaust stack.

- Wastewater streams to a treatment plant.

- Slag Flotation Circuit:
  - Slag slow cooling in ladles breaking, crushing and milling.
  - Slag flotation with related reagent systems.
  - Slag flotation concentrate and tailings dewatering and handling.
  - Slag concentrate re-cycled to DBF; Slag tails disposed to TSF.

Blister and anode-grade copper from the smelter will be sold, for refining by others.

Figure 1.11 Kakula Concentrator Process Block Flow Diagram

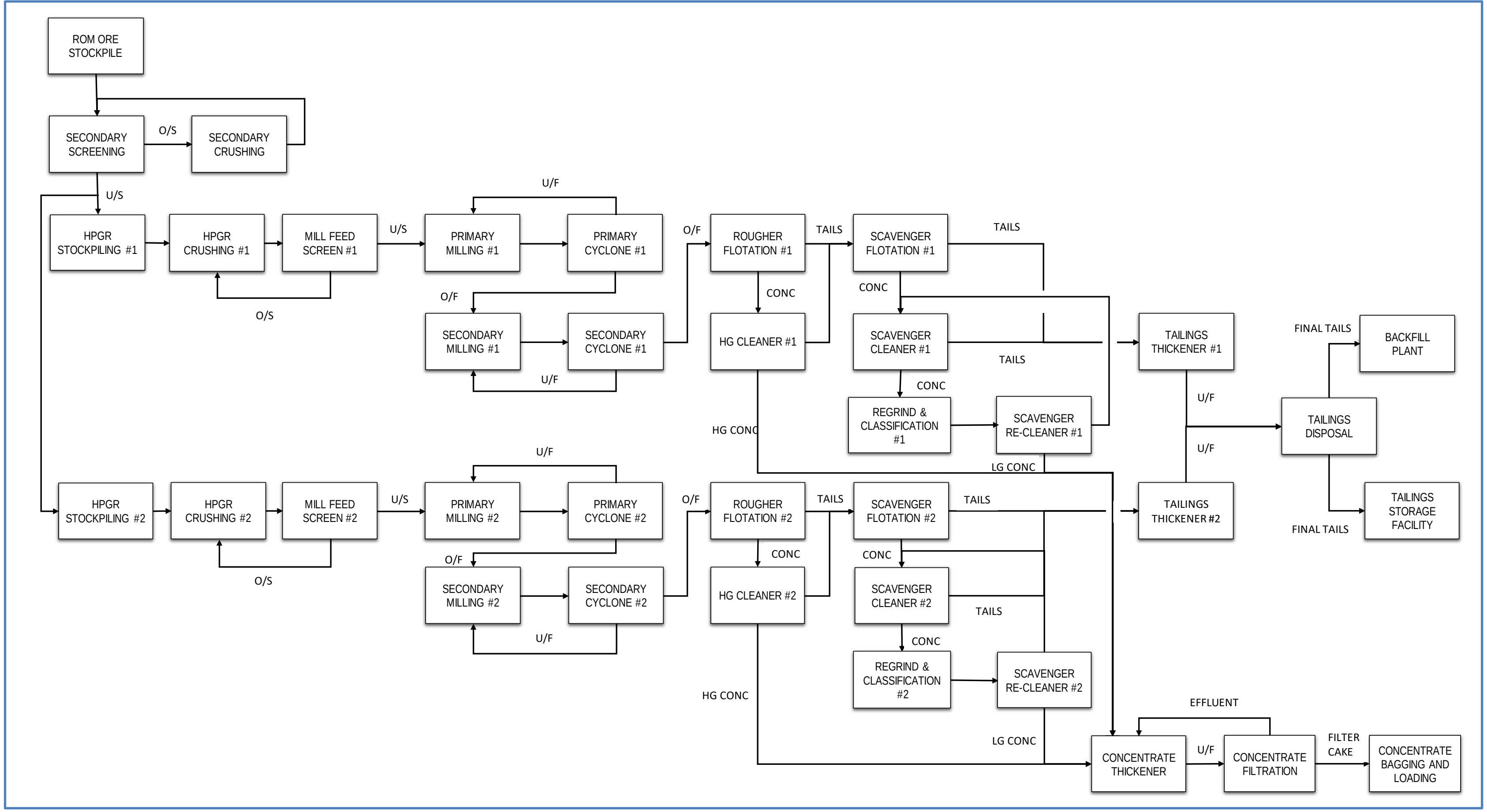


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Figure 1.12 Kamoa Concentrator Process Block Flow Diagram

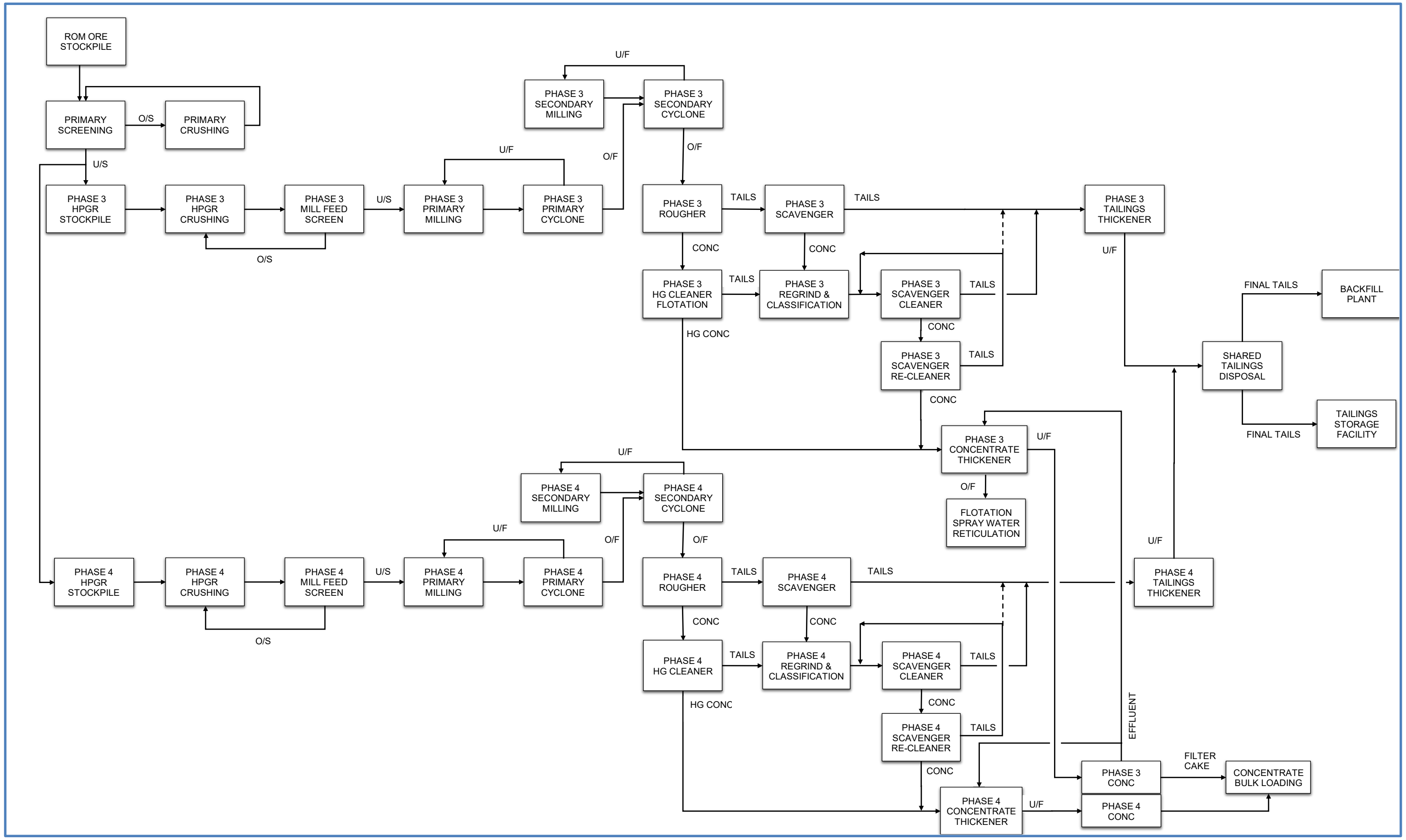

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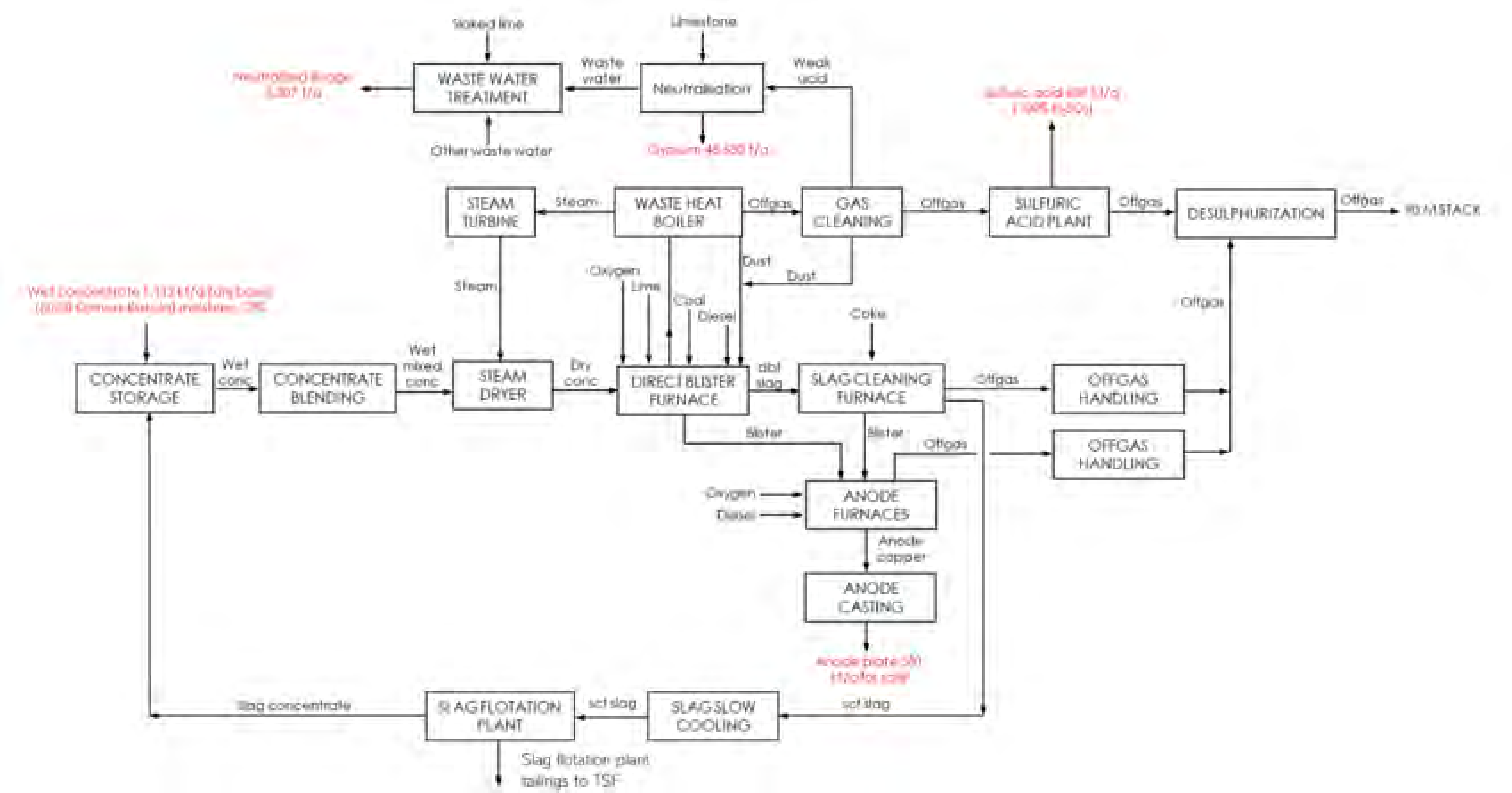
Figure 1.13 Smelter Process Block Flow Diagram



1.16.3 Kamoa-Kakula 2023 PFS Transport

Kamoa-Kakula's concentrate and blister copper products are transported by truck and exported via the ports of Durban in South Africa, and Dar es Salaam in Tanzania, and to a lesser extent, Walvis Bay in Namibia, and Beira in Mozambique.

The Phase 4 expansion in the Kamoa-Kakula IDP23 includes the cost of a railway spur line from Kamoa-Kakula to the main railway line near Kolwezi, which will be used for an additional export route transporting copper products by rail from the mine to the port of Lobito in Angola once operational.

1.16.4 Kamoa-Kakula 2023 PFS Results

The Kamoa-Kakula 2023 PFS analyses a production case with an expansion of the Kamoa-Kakula concentrator processing facilities, and associated infrastructure to 19.2 Mtpa, and includes a smelter, and five separate underground mining operations with associated capital and operating costs.

The five mines are listed below:

- Kakula Mine (9.2 Mtpa).
- Kakula West Mine (6.2 Mtpa).
- Kamoa 1 Mine (6.0 Mtpa).
- Kansoko Sud Mine (2.0 Mtpa).
- Kamoa 2 (6.0 Mtpa).

The combined LOM production scenario schedules 472 Mt to be mined at an average grade of 3.94% copper, producing 37.0 Mt of high-grade copper concentrate, containing approximately 35 billion pounds of copper.

The economic analysis uses a long-term price assumption of US$3.70/lb of copper and returns an after-tax NPV at an 8% discount rate of US$19.1 billion. The Kamoa-Kakula 2023 PFS has a payback period of 1.6-years. The LOM average mine site cash cost is US$1.52/lb of copper and C1 cash cost is US$1.31/lb of copper.

The estimated Phase 3 capital cost, including contingency, is US$3,037M. The estimated Phase 4 capital cost, including contingency, is US$1,553M. The key results of the Kamoa-Kakula 2023 PFS are summarised in Table 1.5, Table 1.6, Table 1.7, Table 1.8, Table 1.9, and Table 1.10 summarises the financial results.

The Process Production schedule and concentrate and metal production is shown in Figure 1.14 (annual cash flow is shown on the left vertical axis, and cumulative cash flow on the right axis) and Figure 1.15, respectively.

Table 1.5 Kamoa-Kakula 2023 PFS Summary

Item	Unit	Total
Total Processed		
Quantity Milled	kt	476,195
Copper Feed Grade	%	3.94
Total Concentrate Produced		
Copper Concentrate Produced	kt (dry)	37,802
Copper Recovery	%	86.62
Copper Concentrate Grade	%	43.05
Contained Metal in Concentrate	Mlb	35,875
Contained Metal in Concentrate	kt	16,273
Annual Average (2023-2024)		
Ore Milled	kt	9,106
Copper Feed Grade	%	5.75
Copper Concentrate Produced	kt (dry)	917
Contained Copper in Concentrate	Mlb	1,004
Contained Copper in Concentrate	kt	455
C1 Cash Cost	US$/lb. payable	1.45
EBITDA	US$M	2,015
Annual Average (2025-2029)		
Ore Milled	kt	14,194
Copper Feed Grade	%	5.30
Copper Concentrate Produced	kt (dry)	1,431
Contained Copper in Concentrate	Mlb	1,442
Contained Copper in Concentrate	kt	654
C1 Cash Cost	US$/lb. payable	1.15
EBITDA	US$M	3,522
Annual Average (First 10-Years)		
Ore Milled	kt	14,428
Copper Feed Grade	%	4.94
Copper Concentrate Produced	kt (dry)	1,379
Contained Copper in Concentrate	Mlb	1,368
Contained Copper in Concentrate	kt	620
C1 Cash Cost	US$/lb. payable	1.22
EBITDA	US$M	3,151
Key Financial Results		
Remaining Phase 3 Capital Costs	US$M	3,037
Phase 4 Capital Costs	US$M	1,553
Sustaining Capital Costs	US$M	5,583
LOM Avg. C1 Cash Cost	US$/lb Payable Cu	1.31
LOM Avg. Total Cash Costs	US$/lb Payable Cu	1.52
LOM Avg. Site Operating Costs	US$/t Milled	72.75
After-Tax NPV8%	US$M	19,062
Project Life	Years	33

Table 1.6 Kamoa-Kakula 2023 PFS Financial Results

	Discount Rate	Before Taxation	After Taxation
Net Present Value (US$M)	Undiscounted	67,966	47,969
	4.0%	41,321	28,966
	6.0%	33,325	23,272
	8.0%	27,407	19,062
	10.0%	22,937	15,884
	12.0%	19,493	13,438
Project Payback Period (Years)	–	1.2	1.6

Table 1.7 Kamoa-Kakula 2023 PFS Production and Processing

Item	Unit	2023-2024	2025-2029	First 10-Years	LOM Average
Total Processed					
Quantity Milled	kt	9,106	14,194	14,428	14,430
Copper Feed Grade	%	5.75	5.30	4.94	3.94
Total Concentrate Produced					
Copper Concentrate Produced	kt (dry)	917	1,431	1,379	1,146
Copper Recovery	%	86.97	87.02	87.02	86.62
Copper Concentrate Grade	% Cu	49.67	45.70	45.01	43.05
Copper in Concentrate					
Contained Copper	Mlb	1,004	1,442	1,368	1,087
Contained Copper	kt	455	654	620	493
Concentrate Smelted / Sold					
Concentrate Smelted (Kamoa)	kt (dry)	–	1,133	936	861
Concentrate Tolled (LCS)	kt (dry)	134	134	134	120
Concentrate Sold	kt (dry)	783	164	310	165
Payable Copper Sold					
Blister Anodes (Kamoa)	kt	–	496	396	353
Blister Copper (LCS)	kt	64	65	63	55
Copper in Concentrate	kt	376	80	147	75
Payable Metal					
Copper	Mlb	971	1,411	1,336	1,064
Copper	kt	440	640	606	483

Note: The 2023-2024 average includes approximately 20 kt of copper in concentrate that is processed by the Phase 3 concentrator during the ramp-up period in 2024.

Table 1.8 Kamoa-Kakula 2023 PFS Capital Costs

Capital Costs (US$M)	Phase 3 Capital US$M	Phase 4 Capital US$M	Sustaining Capital US$M	Total US$M
Underground Mining				
Underground Mining	543	684	2,747	3,974
Mining Mobile Equipment	63	66	1,238	1,367
Subtotal	607	750	3,984	5,341
Power and Smelter				
Smelter Total	906	–	165	1,071
Power Infrastructure	84	134	–	218
Subtotal	990	134	165	1,289
Concentrator and Tailings				
Process Plant	262	238	193	693
Tailings	57	–	404	461
Subtotal	320	238	597	1,154
Infrastructure				
General Surface Infrastructure	662	98	150	910
Rail Spur	–	84	70	154
Subtotal	662	182	220	1,064
Indirects				
EPCM	127	141	5	273
Owners Cost	83	–	15	98
Customs Duties	92	44	175	311
Closure	–	–	145	145
Subtotal	302	185	340	826
Capital Expenditure Before Contingency	2,880	1,488	5,306	9,674
Contingency	157	65	277	499
Capital Expenditure After Contingency	3,037	1,553	5,583	10,173

Totals have been rounded.

Table 1.9 Kamoa-Kakula 2023 PFS Unit Operating Costs

	US$/lb Payable Cu		
	2023-2024	2025-2029	First 10-Years
Mining	0.41	0.44	0.47
Processing	0.16	0.15	0.16
Smelter	–	0.16	0.13
Logistics	0.51	0.24	0.29
Treatment, refining and smelter charges	0.24	0.12	0.14
General and Administration	0.13	0.10	0.09
Sulfuric Acid Credits[1]	–	–0.07	–0.06
C1 Cash Cost	1.45	1.15	1.22
Royalties and Export Tax	0.29	0.21	0.22
Total Cash Cost	1.74	1.36	1.44

[1] Acid Selling Price $150/ t Acid.

Table 1.10 Kamoa-Kakula 2023 PFS Revenue and Operating Costs

	Total LOM	2023-2024	2025-2029	First 10-Years	LOM Average
	US$M	US$/t Milled			
Revenue					
Copper in Blister	110,500	60	333	265	232
Copper in Concentrate	20,569	351	47	85	43
Acid Production	2,618	–	7	6	5
Gross Sales Revenue	133,687	411	386	356	281
Less: Realisation Costs					
Logistics	9,053	55	24	27	19
Treatment, refining and smelter charges	4,719	25	12	13	10
Royalties and Export Tax	7,417	31	21	21	16
Total Realisation Costs	21,189	111	57	60	44
Net Sales Revenue	112,498	300	329	296	236
Site Operating Costs					
UG Mining	19,380	48	40	42	41
Processing	7,167	17	15	15	15
Smelter	5,298	–	16	12	11
General and Administration	2,800	14	10	9	6
Total	34,644	79	81	78	73
EBITDA	77,854	221	248	218	163
EBITDA Margin (%)	58.2%	53.9%	64.3%	61.3%	58.2%

Note: The copper price used in the economic analysis is $3.80/lb. in 2023, $3.90/lb. in 2024, $4.00/lb. in 2025, $4.00/lb. in 2026 and a long-term copper price of $3.70/lb. from 2027 onwards.

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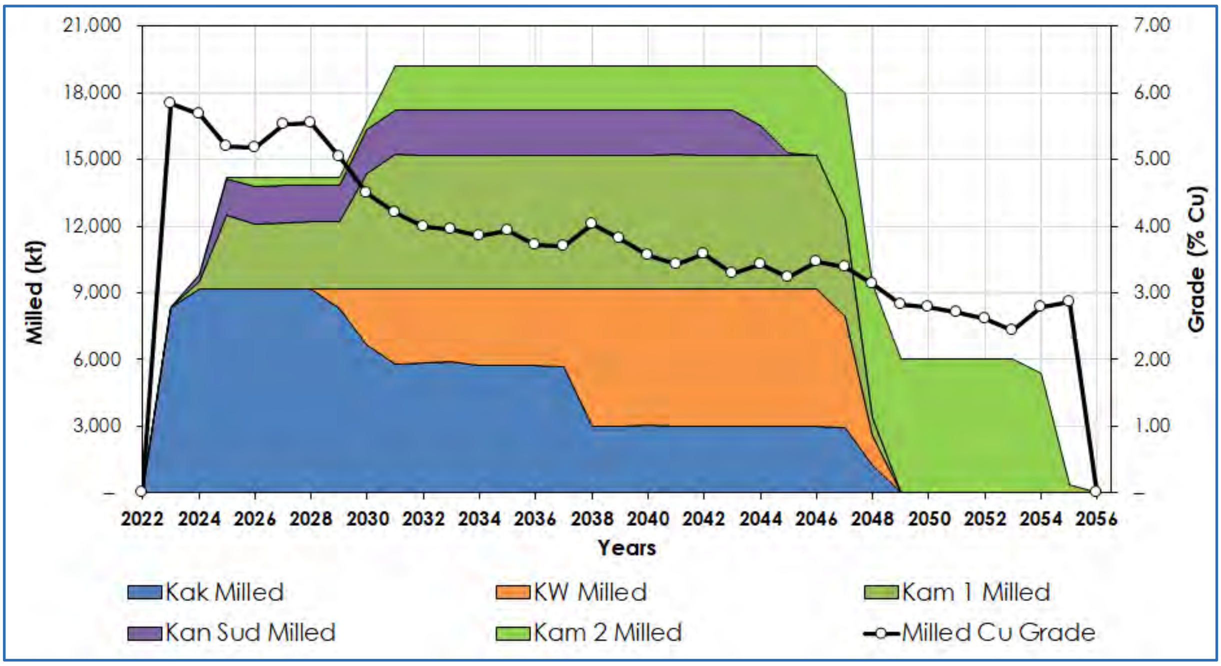
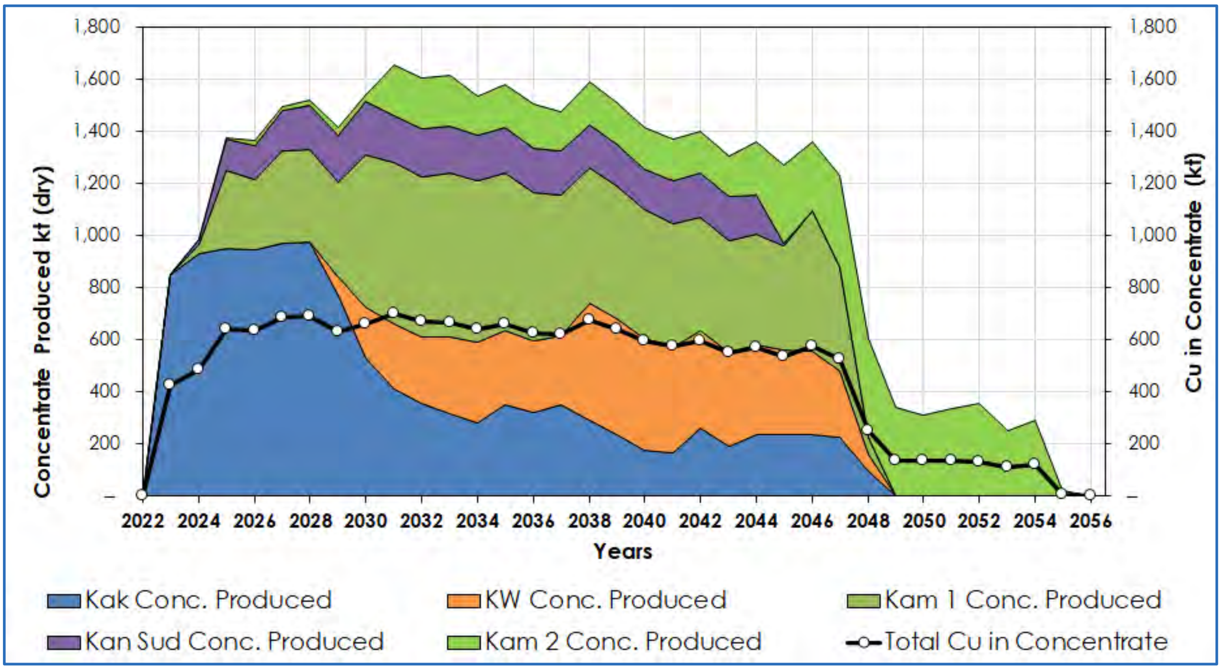
Figure 1.14 Kamoa-Kakula 2023 PFS Process Production



OreWin, 2023.

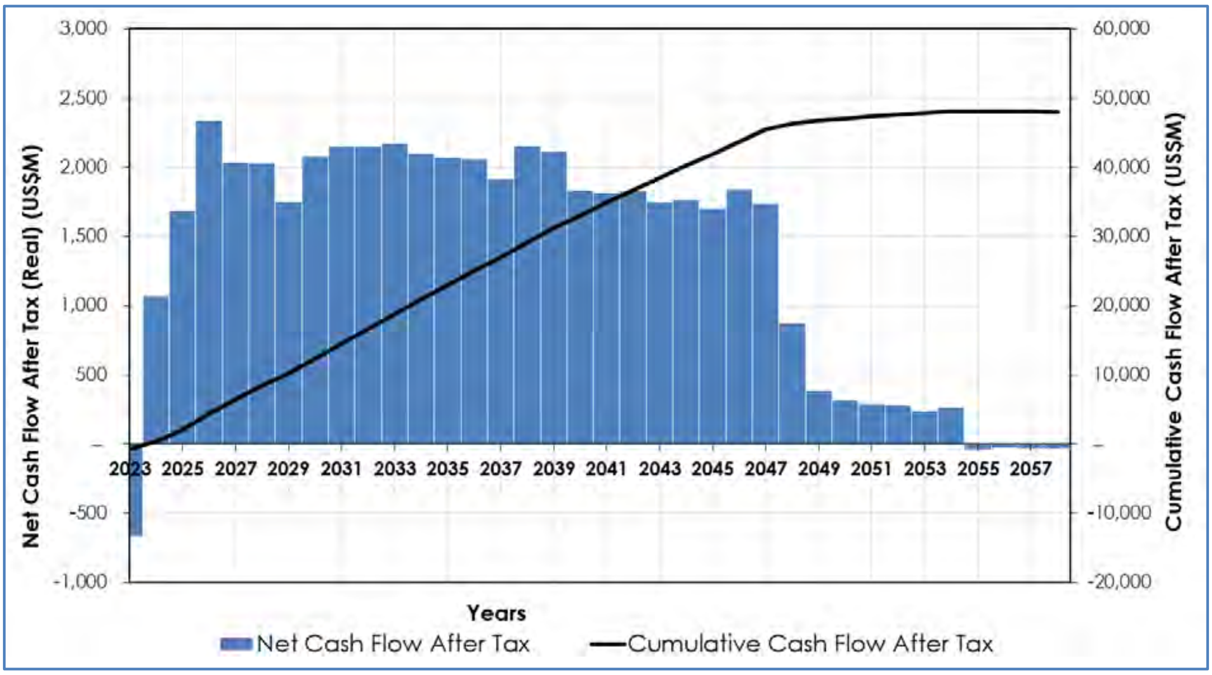
Figure 1.15 Kamoa-Kakula 2023 PFS Concentrate and Metal Production



OreWin, 2023.

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). Kamoa-Kakula 2023 PFS copper price sensitivities are shown in Table 1.11.

Figure 1.16 Kamoa-Kakula 2023 PFS Mine Projected Cumulative Cash Flow



OreWin, 2023.

Table 1.11 Kamoa-Kakula 2023 PFS Copper Price Sensitivity

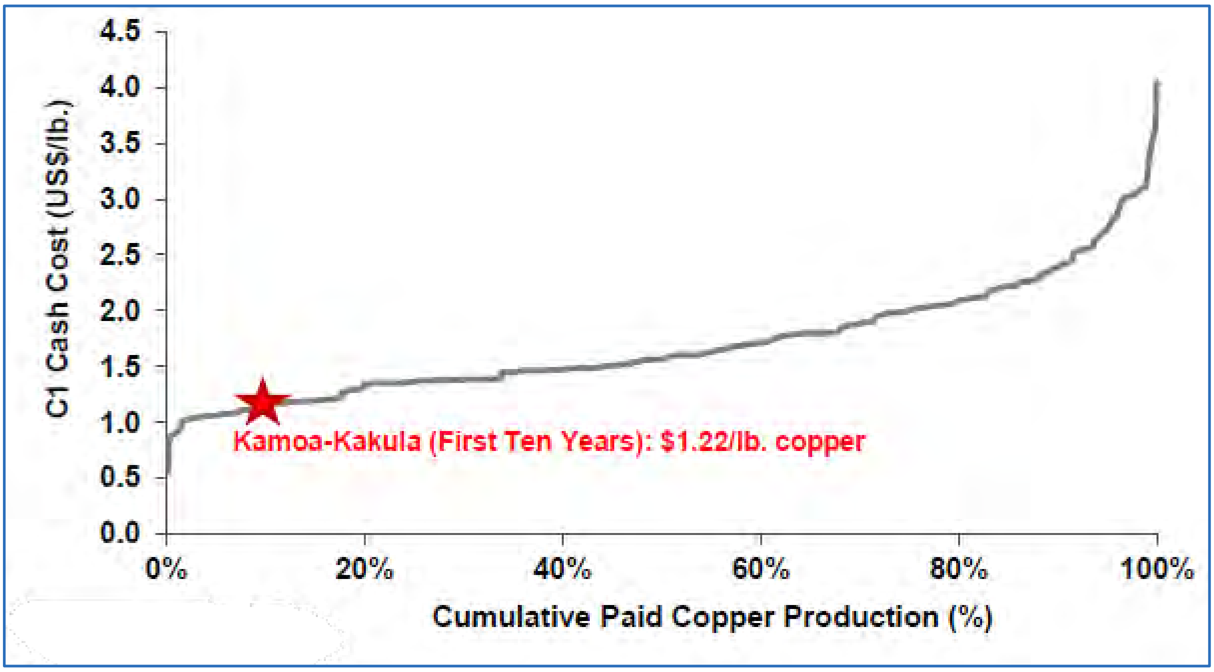
After-Tax NPV (US$M)	Copper Price - US$/lb									
Discount Rate	2.00	2.50	3.00	3.50	3.70	4.00	4.25	4.50	5.00	6.00
Undiscounted	12,760	23,279	33,732	43,902	47,969	54,069	59,153	64,237	72,562	87,982
4.0%	9,004	14,953	20,846	26,646	28,966	32,446	35,346	38,246	42,990	51,776
6.0%	7,734	12,363	16,940	21,463	23,272	25,986	28,248	30,509	34,211	41,069
8.0%	6,733	10,407	14,032	17,625	19,062	21,218	23,015	24,811	27,756	33,213
10.0%	5,934	8,900	11,821	14,723	15,884	17,626	19,077	20,528	22,910	27,328
12.0%	5,285	7,717	10,107	12,486	13,438	14,865	16,055	17,244	19,201	22,833
15.0%	4,519	6,369	8,183	9,992	10,715	11,800	12,704	13,609	15,101	17,875

Note: The copper price used in the economic analysis is $3.80/lb. in 2023, $3.90/lb. in 2024, $4.00/lb. in 2025, $4.00/lb. in 2026, and a long-term copper price of $3.70/lb. from 2027 onwards.

Figure 1.17 shows the C1 pro-rata copper cash costs of the Kamoa-Kakula 2023 PFS on Wood Mackenzie's industry cost curve. This figure represents C1 pro-rata cash costs that reflect the direct cash costs of producing paid copper incorporating mining, processing, smelting, logistics and off-site realisation costs, having made appropriate allowance for the costs associated with the co-product revenue streams.

The Kamoa-Kakula 2023 PFS was not reviewed by Wood Mackenzie prior to filing, and information was sourced from public disclosures.

Figure 1.17 2025 C1 Pro-rata Copper Cash Costs



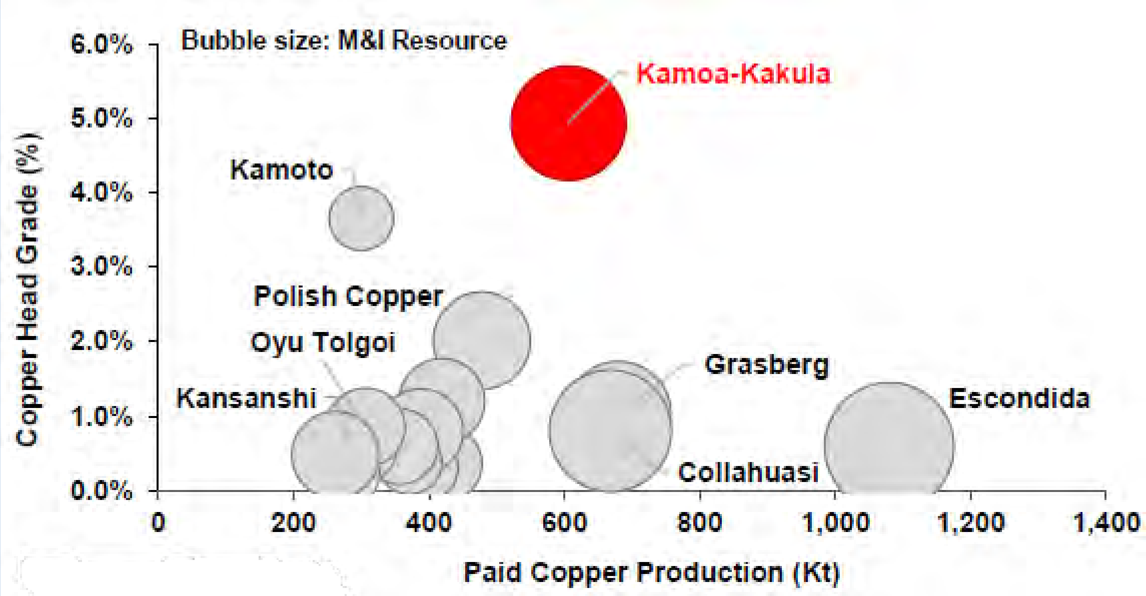
Ivanhoe, 2023. Source: Wood Mackenzie 2023.

Figure 1.18 compares the reported copper production in 2025 for the 20 highest producers by paid copper production. The Kamoa-Kakula 2023 PFS production and grade are based on average paid copper production and average copper feed grade during the first 10-years.

Figure 1.19 shows projected top 10 largest copper projects in 2025. The 'Copper Head Grade' for the projects benchmarked by Wood Mackenzie reflects the average reserve grade. The estimates are based on public disclosure and information gathered by Wood Mackenzie.

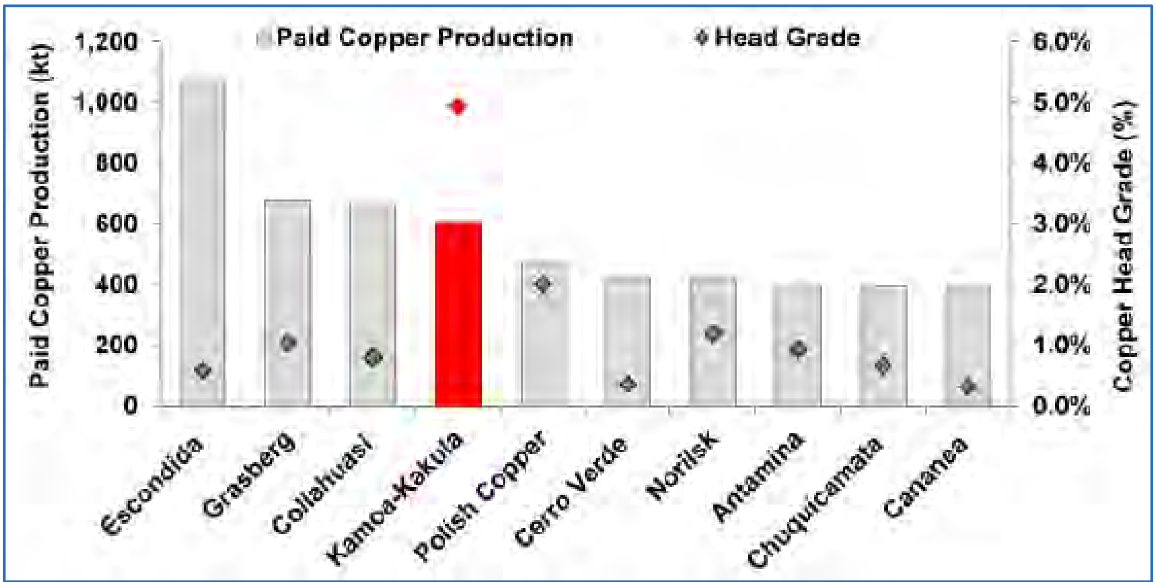
The Kamoa-Kakula IDP23 was not reviewed by Wood Mackenzie prior to filing.

Figure 1.18 2025 Predicted World Copper Producer Production



Ivanhoe, 2023. Source: Wood Mackenzie, 2023.

Figure 1.19 World Copper Producer Production and Head Grade



Ivanhoe, 2023. Source: Wood Mackenzie, 2023.

1.17 Kamoa-Kakula 2023 PEA

The Kamoa-Kakula 2023 PEA analyses a production case of maintaining overall production rate of up to 19.2 Mtpa from the Kamoa-Kakula 2023 PFS with the addition of four new underground mines in the Kamoa area (called, Kamoa 3, 4, 5, and 6). The additional four mines will extend the Kamoa-Kakula Copper Complex mine life by nine-years. Overview of deposits included within the Kamoa-Kakula 2023 PEA (outlined by green dotted line) is shown in Figure 1.20.

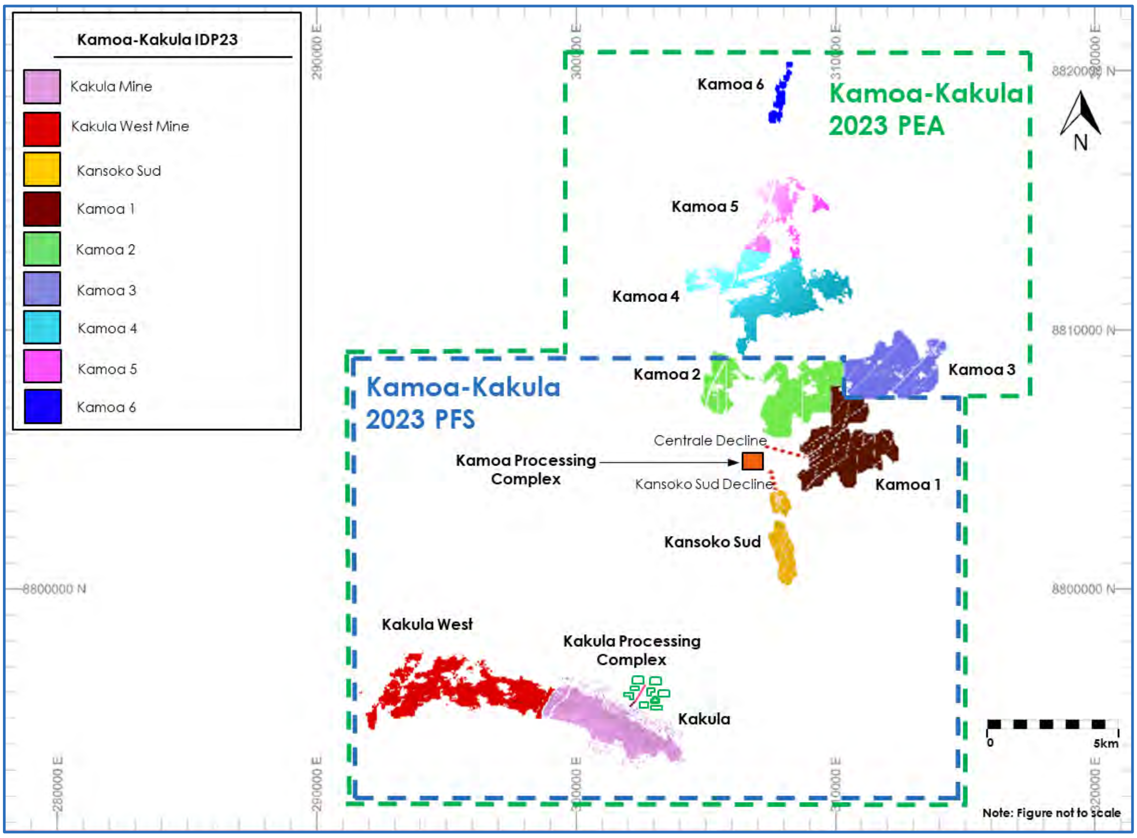
The nine mines are listed below:

- Kakula Mine (PFS 9.2 Mtpa).
- Kakula West Mine (PFS 6.2 Mtpa).
- Kamoa 1 Mine (PFS 6.0 Mtpa).
- Kansoko Sud Mine (PFS 2.0 Mtpa).
- Kamoa 2 (PFS 6.0 Mtpa).
- Kamoa 3 (PEA 6.0 Mtpa).
- Kamoa 4 (PEA 6.0 Mtpa).
- Kamoa 5 (PEA 3.0 Mtpa).
- Kamoa 6 (PEA 1.0 Mtpa).

The Kamoa-Kakula 2023 PEA as part of the Kamoa-Kakula IDP23 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 2023 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 2023 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.

Figure 1.20 Kamoa-Kakula 2023 PEA Mining Locations



OreWin, 2023.

1.17.1 Kamoa-Kakula 2023 PEA Mining

The Kamoa-Kakula 2023 PEA assesses a nine-year mine life extension of the Kamoa-Kakula Copper Complex, in addition to the Kamoa-Kakula 2023 PFS. This case includes the addition of four new underground mines in the Kamoa area (called Kamoa 3, 4, 5, and 6) and additional capital, and operating costs to bring plant feed from the Kamoa mines to the Kakula concentrator by overland conveyors. This maintains the overall production rate of up to 19.2 Mtpa.

The Kamoa-Kakula 2023 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.

The primary mining method for the Kamoa-Kakula deposits (drift-and-fill), was selected based on the results of a trade-off study which compared six different variations of drift-and-fill mining to identify the most suitable. The drift-and-fill method will be utilised as the primary mining method for all Kamoa-Kakula deposits. Identified mining areas with a dip greater than 25° will be mined using the hanging wall access drift-and-fill (HWAD&F) method. To establish the drift-and-fill mining method, a pair of twin perimeter declines are driven at a defined offset to the extremities of each deposit. Twin connection drifts are then developed across the target orebody. The selected drift-and-fill mining methods are explained in detail in Section 16.2.

1.17.2 Kamoa-Kakula 2023 PEA Processing and Infrastructure

The Kamoa-Kakula 2023 PEA processing facilities are as per the Kamoa-Kakula 2023 PFS and considers an increase in production capacity from 7.6 Mtpa up to a total of 19.2 Mtpa. This increase in production will be achieved by debottlenecking of the Kakula concentrator facility to 9.2 Mtpa, and the phased addition of a further 10.0 Mtpa processing facility located at Kamoa. The Kamoa-Kakula 2023 PEA has processing, smelting, and infrastructure facilities that include:

- A 9.2 Mtpa processing facility, complete with surface crushing and screening, milling, and flotation, consisting of two 4.6 Mtpa concentrator streams, a smelter, and associated infrastructure located at the Kakula Mine area.
- A 10.0 Mtpa processing facility, complete with surface crushing and screening, milling, and flotation, consisting of two 5.0 Mtpa concentrator streams and associated infrastructure located at the Kamoa Mine area.
- All associated infrastructure included in the Kamoa-Kakula 2023 PFS, with the addition of an overland conveyor from the Kamoa mines to the Kakula processing facility once Kakula and Kakula West mines are depleted, and 19.2 Mtpa production is sustained by Kamoa mines.
- Smelter as per PFS to be operational at reduced capacity due to the feed material sourced from Kamoa mines only.

1.17.3 Kamoa-Kakula 2023 PEA Results

The Kamoa-Kakula 2023 PEA analyses a production case of maintaining overall production rate of up to 19.2 Mtpa from the Kamoa-Kakula 2023 PFS with the addition of four new underground mines in the Kamoa area (called, Kamoa 3, 4, 5, and 6). The additional four mines will extend the Kamoa-Kakula Copper Complex mine life by nine-years. The nine mines, including four additional PEA mines are listed below:

- Kakula Mine (PFS 9.2 Mtpa).
- Kakula West Mine (PFS 6.2 Mtpa).
- Kamoa 1 Mine (PFS 6.0 Mtpa).
- Kansoko Sud Mine (PFS 2.0 Mtpa).
- Kamoa 2 (PFS 6.0 Mtpa).
- Kamoa 3 (PEA 6.0 Mtpa).
- Kamoa 4 (PEA 6.0 Mtpa).
- Kamoa 5 (PEA 3.0 Mtpa).
- Kamoa 6 (PEA 1.0 Mtpa).

The combined LOM production scenario schedules 653 Mt to be mined at an average grade of 3.70% copper, producing 50.8 Mt of high-grade copper concentrate, containing approximately 45.7 billion pounds of copper.

The economic analysis uses a long-term price assumption of US$3.70/lb of copper and returns an after-tax NPV at an 8% discount rate of US$20.2 billion. The Kamoa-Kakula 2023 PEA has a payback period of 1.6-years. The LOM average mine site cash cost is US$1.53/lb of copper and C1 cash cost is US$1.32/lb of copper.

The estimated Phase 3 capital cost, including contingency, is US$3,037M. The estimated Phase 4 capital cost, including contingency, is US$1,553M. The key results of the Kamoa-Kakula 2023 PEA are summarised in Table 1.12, Table 1.13, Table 1.14, Table 1.15, Table 1.16 and Table 1.17 summarises the financial results.

The Process Production schedule and concentrate and metal production are shown in Figure 1.14 (annual cash flow is shown on the left vertical axis and cumulative cash flow on the right axis) and Figure 1.15, respectively.

Table 1.12 Kamoa-Kakula 2023 PEA Summary

Item	Unit	Total
Total Processed		
Quantity Milled	kt	657,428
Copper Feed Grade	%	3.70
Total Concentrate Produced		
Copper Concentrate Produced	kt (dry)	50,761
Copper Recovery	%	86.45
Copper Concentrate Grade	%	41.45
Contained Metal in Concentrate	Mlb	46,384
Contained Metal in Concentrate	kt	21,040
Annual Average (2023-2024)		
Ore Milled	kt	9,106
Copper Feed Grade	%	5.75
Copper Concentrate Produced	kt (dry)	917
Contained Copper in Concentrate	Mlb	1,004
Contained Copper in Concentrate	kt	455
C1 Cash Cost	US$/lb. payable	1.45
EBITDA	US$M	2,015
Annual Average (2025-2029)		
Ore Milled	kt	14,194
Copper Feed Grade	%	5.30
Copper Concentrate Produced	kt (dry)	1,431
Contained Copper in Concentrate	Mlb	1,442
Contained Copper in Concentrate	kt	654
C1 Cash Cost	US$/lb. payable	1.15
EBITDA	US$M	3,522
Annual Average (First 10-Years)		
Ore Milled	kt	14,428
Copper Feed Grade	%	4.94
Copper Concentrate Produced	kt (dry)	1,379
Contained Copper in Concentrate	Mlb	1,368
Contained Copper in Concentrate	kt	620
C1 Cash Cost	US$/lb. payable	1.22
EBITDA	US$M	3,151
Key Financial Results		
Remaining Phase 3 Capital Costs	US$M	3,037
Phase 4 Capital Costs	US$M	1,553
Sustaining Capital Costs	US$M	8,858
LOM Avg. C1 Cash Cost	US$/lb Payable Cu	1.32
LOM Avg. Total Cash Costs	US$/lb Payable Cu	1.53
LOM Avg. Site Operating Costs	US$/t Milled	70.57
After-Tax NPV8%	US$M	20,224
Project Life	Years	42

Table 1.13 Kamoa-Kakula 2023 PEA Financial Results

	Discount Rate	Before Taxation	After Taxation
Net Present Value (US$M)	Undiscounted	86,453	60,760
	4.0%	46,742	32,708
	6.0%	36,329	25,342
	8.0%	29,096	20,224
	10.0%	23,899	16,544
	12.0%	20,049	13,818
Project Payback Period (Years)	–	1.2	1.6

Table 1.14 Kamoa-Kakula 2023 PEA Production and Processing

Item	Unit	2023-2024	2025-2029	First 10-Years	LOM Average
Total Processed					
Quantity Milled	kt	9,106	14,194	14,428	15,653
Copper Feed Grade	%	5.75	5.30	4.94	3.70
Total Concentrate Produced					
Copper Concentrate Produced	kt (dry)	917	1,431	1,379	1,209
Copper Recovery	%	86.97	87.02	87.02	86.45
Copper Concentrate Grade	% Cu	49.67	45.70	45.01	41.45
Copper in Concentrate					
Contained Copper	Mlb	1,004	1,442	1,368	1,104
Contained Copper	kt	455	654	620	501
Concentrate Smelted / Sold					
Concentrate Smelted (Kamoa)	kt (dry)	–	1,133	936	945
Concentrate Tolled (LCS)	kt (dry)	134	134	134	106
Concentrate Sold	kt (dry)	783	164	310	158
Payable Copper Sold					
Blister Anodes (Kamoa)	kt	–	496	396	374
Blister Copper (LCS)	kt	64	65	63	47
Copper in Concentrate	kt	376	80	147	69
Payable Metal					
Copper	Mlb	971	1,411	1,336	1,081
Copper	kt	440	640	606	490

Note: The 2023-2024 average includes approximately 20 kt of copper in concentrate that is processed by the Phase 3 concentrator during the ramp-up period in 2024.

Table 1.15 Kamoa-Kakula 2023 PEA Capital Costs

Capital Costs (US$M)	Phase 3 Capital US$M	Phase 4 Capital US$M	Sustaining Capital US$M	Total US$M
Underground Mining				
Underground Mining	543	684	4,922	6,149
Mining Mobile Equipment	63	66	1,735	1,864
Subtotal	607	750	6,657	8,013
Power and Smelter				
Smelter Total	906	–	215	1,121
Power Infrastructure	84	134	–	218
Subtotal	990	134	215	1,339
Concentrator and Tailings				
Process Plant	262	238	307	807
Tailings	57	–	534	591
Subtotal	320	238	841	1,398
Infrastructure				
General Surface Infrastructure	662	98	236	997
Rail Spur	–	84	95	179
Subtotal	662	182	406	1,250
Indirects				
EPCM	127	141	5	273
Owners Cost	83	–	15	98
Customs Duties	92	44	260	396
Closure	–	–	145	145
Subtotal	302	185	425	912
Capital Expenditure Before Contingency	2,880	1,488	8,544	12,912
Contingency	157	65	314	536
Capital Expenditure After Contingency	3,037	1,553	8,858	13,448

Totals have been rounded.

Table 1.16 Kamoa-Kakula 2023 PEA Unit Operating Costs

	Payable Copper (US$/lb)			
	2023-2024	2025-2029	First 10-Years	LOM Average
Mining	0.41	0.44	0.47	0.58
Processing	0.16	0.15	0.16	0.21
Smelter	–	0.16	0.13	0.17
Logistics	0.51	0.24	0.29	0.26
Treatment, refining and smelter charges	0.24	0.12	0.14	0.13
General and Administration	0.13	0.10	0.09	0.07
Sulfuric Acid Credits[1]	–	–0.07	-0.06	–0.09
C1 Cash Cost	1.45	1.15	1.22	1.32
Royalties and Export Tax	0.29	0.21	0.22	0.21
Total Cash Cost	1.74	1.36	1.44	1.53

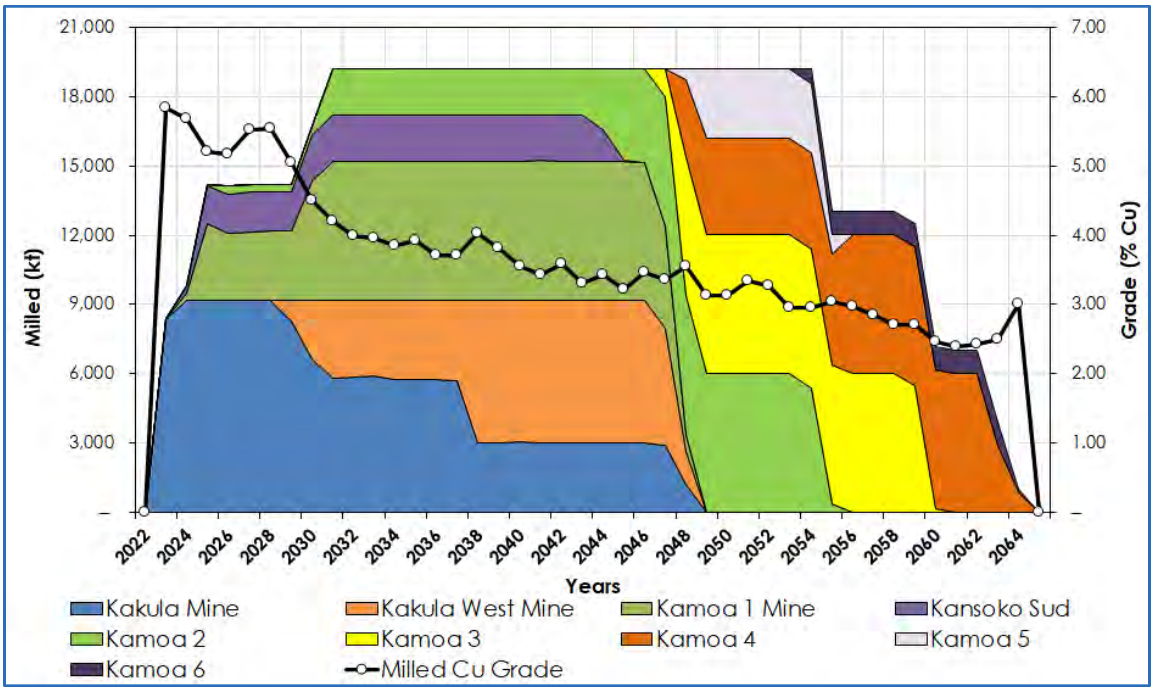
[1] Acid Selling Price $150/ t Acid.

Table 1.17 Kamoa-Kakula 2023 PEA Revenue and Operating Costs

	Total LOM	2023-2024	2025-2029	First 10-Years	LOM Average
	US$M	US$/t Milled			
Revenue					
Copper in Blister	145,091	60	333	265	221
Copper in Concentrate	23,967	351	47	85	36
Acid Production	3,912	–	7	6	6
Gross Sales Revenue	172,970	411	386	356	263
Less: Realisation Costs					
Logistics	11,585	55	24	27	18
Treatment, refining and smelter charges	5,816	25	12	13	9
Royalties and Export Tax	9,572	31	21	21	15
Total Realisation Costs	26,972	111	57	60	41
Net Sales Revenue	145,998	300	329	296	222
Site Operating Costs					
UG Mining	26,274	48	40	42	40
Processing	9,409	17	15	15	14
Smelter	7,724	–	16	12	12
General and Administration	2,985	14	10	9	5
Total	46,393	79	81	78	71
EBITDA	99,606	221	248	218	152
EBITDA Margin (%)	57.6%	53.9%	64.3%	61.3%	57.6%

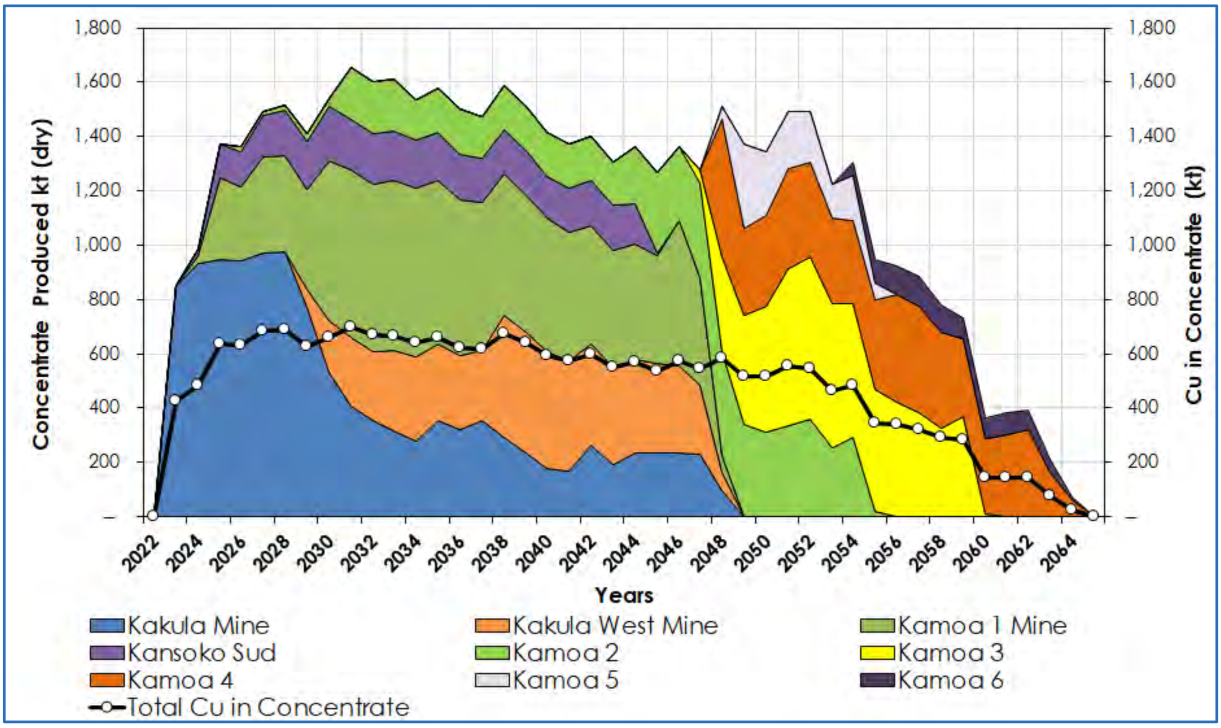
Note: The copper price used in the economic analysis is $3.80/lb. in 2023, $3.90/lb. in 2024, $4.00/lb. in 2025, $4.00/lb. in 2026 and a long-term copper price of $3.70/lb. from 2027 onwards.
Totals have been rounded.

Figure 1.21 Kamoa-Kakula 2023 PEA Process Production



OreWin, 2023.

Figure 1.22 Kamoa-Kakula 2023 PEA Concentrate and Metal Production



OreWin, 2023.

The after-tax net present value (NPV) sensitivity to metal price variation is shown in Table 1.18 for copper prices from US$2.00–US$6.00/lb.

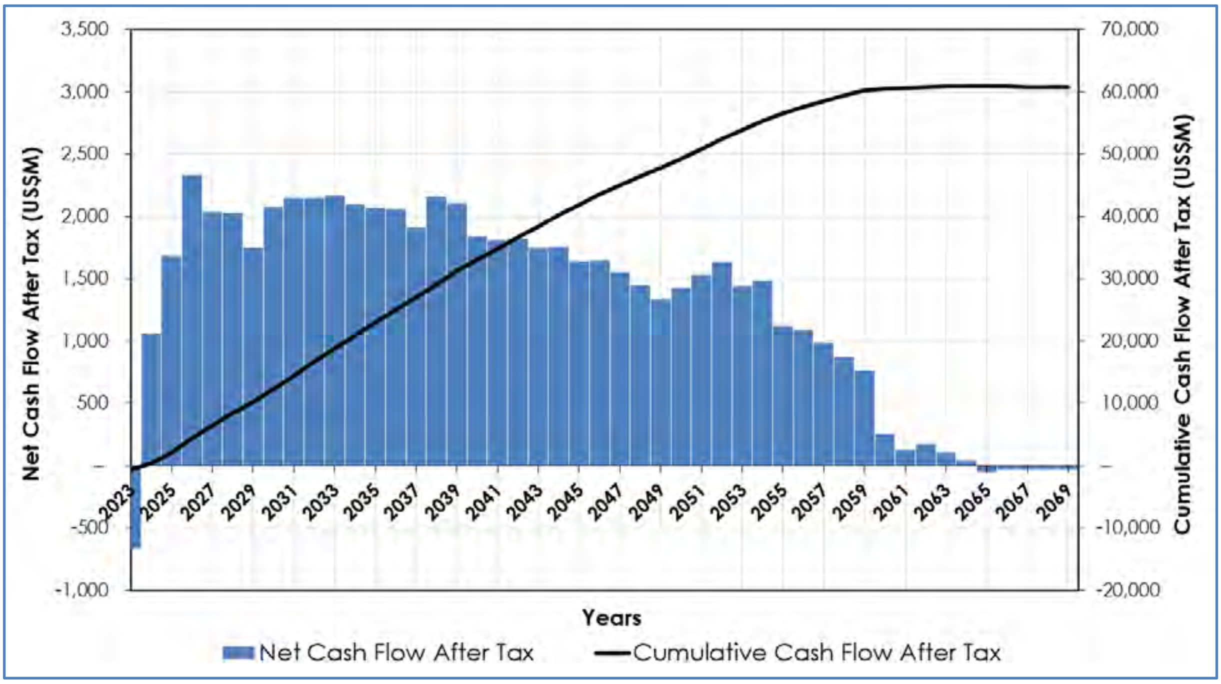
The annual and cumulative cash flows are shown in Figure 1.23 (annual cash flow is shown on the left vertical axis and cumulative cash flow on the right axis).

Table 1.18 Kamoa-Kakula 2023 PEA Copper Price Sensitivity

After-Tax NPV (US$M)	Copper Price (US$/lb)										
Discount Rate	2.00	2.50	3.00	3.50	3.70	4.00	4.25	4.50	5.00	5.50	6.00
Undiscounted	13,765	28,112	41,731	55,323	60,760	68,915	75,710	82,506	93,703	104,102	114,500
4.0%	9,315	16,385	23,190	29,989	32,708	36,787	40,187	43,586	49,149	54,300	59,451
6.0%	7,902	13,155	18,235	23,312	25,342	28,388	30,927	33,465	37,616	41,459	45,301
8.0%	6,822	10,849	14,756	18,661	20,224	22,567	24,520	26,472	29,667	32,625	35,583
10.0%	5,980	9,149	12,231	15,312	16,544	18,393	19,934	21,474	23,998	26,337	28,675
12.0%	5,308	7,859	10,342	12,825	13,818	15,308	16,549	17,791	19,829	21,719	23,609
15.0%	4,526	6,431	8,287	10,143	10,885	11,998	12,926	13,854	15,383	16,803	18,223

Note: The copper price used in the economic analysis is $3.80/lb. in 2023, $3.90/lb. in 2024, $4.00/lb. in 2025, $4.00/lb. in 2026, and a long-term copper price of $3.70/lb. from 2027 onwards.

Figure 1.23 Kamoa-Kakula 2023 PEA Projected Cumulative Cash Flow



OreWin, 2023.

1.18 Interpretation and Conclusions

1.18.1 Mineral Resource Estimate

Mineral Resources for the Project have been estimated using core drill data, have been performed using industry best practices (CIM, 2019), and conform to the requirements of CIM Definition Standards (2014).

MSA has checked the data and estimation methodology used to construct the resource models (Datamine macros) and has validated the resource models. MSA finds the Kamoa and Kakula resource models to be suitable to support Pre-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 programmes 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.
  - 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.
- 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.
- Commodity prices and exchange rates.
- Cut-off grades.

1.18.2 Kamoa-Kakula IDP23

The Kamoa-Kakula IDP23 includes an update of the Kakula and Kamoa Mineral Reserve and updates of the preliminary economic assessment (PEA), including analysis of the Kamoa 3, Kamoa 4, Kamoa 5, and Kamoa 6 Mineral Resource.

The Kamoa-Kakula 2023 PFS has identified a Mineral Reserve and development path that has confirmed significant value in the Kamoa and Kakula deposits.

The Kamoa-Kakula 2023 PFS indicates the combined Kakula, Kakula West, Kansoko Sud, Kamoa 1, and Kamoa 2 mine plans using the 9.2 Mtpa Kakula processing complex, and 10.0 Mtpa Kamoa processing complex, including on-site smelting, generates significant value.

The analysis in the Kamoa-Kakula 2023 PEA assesses a nine-year mine life extension of the Kamoa-Kakula Copper Complex, in addition to the Kamoa-Kakula 2023 PFS. This case includes the addition of four new underground mines in the Kamoa area (called Kamoa 3, 4, 5, and 6) to maintain the overall production rate of up to 19.2 Mtpa.

1.18.3 Kamoa-Kakula IDP23 Engineering and Cost Estimation

Concentrator plant engineering and cost estimates were completed to a higher degree of detail and accuracy as required by PFS due to the availability of the current construction work ongoing and detail Phase 3 Basic engineering study that was completed prior to the Kamoa-Kakula 2023 PFS.

Typically, a PFS will only include reasonably advanced process and mechanical designs and it is normal for a lot of the other design disciplines to be preliminary for a PFS. The China Nerin Engineering Co. Ltd ("Nerin") design documentation was confirmed to be technically adequate for a PFS in all aspects. Some of the documentation had been prepared to a much greater technical extent, more reflective of a definitive feasibility study or for project implementation.

1.19 Recommendations

1.19.1 Further Assessment

The Kamoa-Kakula 2023 PFS described development scenarios for Kamoa, and Kakula, deposits that expand on the initial two phases of the project. The first, Phase 3 is already significantly advanced and is to be followed by Phase 4 for a total processing capacity of 19.2 Mtpa. A holistic approach should be undertaken to optimise the project value. The two additional phases identified in the Kamoa-Kakula 2023 PFS provide the current development plan for the Kamoa-Kakula Mineral Resources. As development continues each stage of the project should be analysed and redefined.

The key areas for further studies are:

- The Kamoa-Kakula 2023 PFS has identified a significant Mineral Reserve to be mined. Performance at the operation has successfully achieved production rates above those planned in the previous studies. The next study work should consider the timing and implementation of the Phase 4 project which is earmarked to be brought on-line once the Phase 3 project reaches steady state. The next study work should consider optimisation for increased production at rates above 20 Mtpa from the combined Kakula and Kamoa mines.
- At Kamoa the groundwork to support bringing forward the timing of Phase 4 should be conducted. This should include further exploration drilling and upgrading of the Mineral Resources and targeted technical studies to support the Phase 4 project implementation.
- The Kamoa-Kakula 2023 PEA indicates that there is potential value in extending the life of the Kamoa-Kakula operation with the addition of Kamoa 3, Kamoa 4, Kamoa 5, and Kamoa 6 mines. To identify this potential, further study will be needed. These studies will be undertaken using a holistic approach into the long-term options to maximise the efficient extraction of the Kamoa-Kakula Mineral Resources.

1.19.2 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, and planned, mine development, and to define the edges of the higher-grade material.

1.19.3 Processing Plant

Extensive metallurgical testwork has 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.

1.19.4 Smelter

Solar power with battery storage may be an alternative to emergency diesel generators to be considered in a study that examines battery capacity, and the importance of uninterrupted power supply for the smelter. It is accepted that use of burnt lime flux in the smelter instead of limestone will reduce power consumption. However, the hazards to operation and maintenance personnel of using burnt lime, and the potential consequences of the marked exothermic reaction if the material is wettened must be considered carefully in the detailed design, and operating practices to be developed for the smelter.

Vendor project references for the combination of diesel burners and supplementary pulverised coal in a controlled flash smelting or DBF operation will provide more confidence if they can be obtained.

The mass balance, as well as the number of equipment items in slag flotation, and product handling, should be investigated further to ensure that product grade objectives, recovery targets, and plant availability will all be achieved.

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 focussed 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 land package of 2,400 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 IDP23 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 Mutshatsha territory in the 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 IDP23:

- OreWin Pty Ltd.: Overall Report preparation, Kamoa-Kakula 2023 PEA analyses, underground mining, Kamoa-Kakula combined production schedules, and financial models.
- The MSA Group: Geology, drillhole data validation, and Mineral Resource estimation for Kamoa and Kakula.
- SRK Consulting South Africa (Pty) Ltd.: PFS Mine geotechnical recommendations.
- 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.
- 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:

Qualified Persons:

- Bernard Peters, B. Eng. (Mining), FAusIMM (201743), employed by OreWin as Technical Director - Mining was responsible for: Sections 1.1 to 1.6, 1.15 to 1.16.1, 1.16.3 to 1.17.1, 1.17.3, 1.18, 1.18.2, 1.19, 1.19.1; Section 2; Section 3; Section 4; Section 5; Section 6; Section 15; Section 16.2 to 16.3.7, 16.3.8.1, 16.3.8.2, 16.3.9, 16.4 to 16.4.7, 16.4.8.1, 16.4.8.2, 16.4.9, 16.5 to 16.5.7, 16.5.8.1, 16.5.8.2, 16.5.9, 16.6 to 16.6.7, 16.6.8.1, 16.6.8.2, 16.6.9, 16.7 to 16.7.7, 16.7.8.1, 16.7.8.2, 16.7.9; Section 19; Section 20; Sections 21.1 to 21.2.1, 21.3; Section 22; Section 23; Sections 24.1 to 24.4, 24.6; Sections 25.2, 25.3; Sections 26.1, 26.3; Section 27.
- Jeremy Witley, Pr.Sci.Nat., SACNASP, FGSSA, HOD, Mineral Resources, The MSA Group, was responsible for: Sections 1.7 to 1.11, 1.13, 1.14, 1.18.1, 1.19, 1.19.2; Section 2; 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 1.19; Section 2; Section 10.6; Section 16.1; Section 27.
- Curtis Smith, B. Eng. (Mining), MAusIMM(CP) (311458), employed by OreWin as Principal Mining Engineer, was responsible for: Sections 1.1 to 1.6, 1.15 to 1.16.1, 1.16.3 to 1.17.1, 1.17.3, 1.18, 1.18.2, 1.19, 1.19.1; Section 2; Section 3; Section 4; Section 5; Section 6; Section 15; Section 16.2 to 16.3.7, 16.3.8.1, 16.3.8.2, 16.3.9, 16.4 to 16.4.7, 16.4.8.1, 16.4.8.2, 16.4.9, 16.5 to 16.5.7, 16.5.8.1, 16.5.8.2, 16.5.9, 16.6 to 16.6.7, 16.6.8.1, 16.6.8.2,16.6.9, 16.7 to 16.7.7, 16.7.8.1, 16.7.8.2, 16.7.9; Section 19; Section 20; Sections 21.1 to 21.2.1, 21.3; Section 22; Section 23; Sections 24.1 to 24.4, 24.6; Sections 25.2, 25.3; Sections 26.1, 26.3; Section 27.
- Marius Phillips, MAusIMM (CP) (227570), Senior Principal Consultant – Mineral Processing, employed by Stantec Australia Pty Ltd, was responsible for: Sections 1.12, 1.16.2, 1.17.2, 1.18.3, 1.19, 1.19.3, 1.19.4; Section 2; Section 10.7; Section 11.3; Section 13; Section 17; Section 24.5; Section 25.4; Section 26.4, 26.5; Section 27.

- Alwyn Scholtz, B.Eng., MSc, Pr.Eng, ECSA (20150110) employed by DRA Global as Study Manager, was responsible for: Section 1.19; Section 2; Sections 16.3.8.3 to 16.3.8.10, 16.4.8.3 to 16.4.8.10, 16.5.8.3 to 16.5.8.10, 16.6.8.3 to 16.6.8.11, 16.7.8.3 to 16.7.8.9; Sections 18.1 to 18.1.6, 18.1.8 to 18.2.3; Sections 21.1, 21.2.2 to 21.3; Section 25.5; Section 26.6; Section 27.
- Andreas Savvas, BSc Eng (Civil), MSc Eng, M.ASCE, MSAIMM, MIMMM (691401) employed by Epoch Resources as Project Director, was responsible for: Section 1.19; Section 2; Sections 18.1.7, 18.2.4; 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, from 6–8 August 2018, from 7-14 March 2022, from 10–17 June 2022, from 9-19 August 2022, and from 30 October – 4 November 2022. The site visits included briefings from KCSA and Ivanhoe Mine Ltd., geology and exploration personnel, exploration drill site inspection, underground inspections of the Kakula, Kansoko and Kamoa 1 mines, site inspections of the Kamoa 1 decline portal and box cut and the sites for future mining, plant and infrastructure, discussions with other QPs and review of the existing infrastructure and facilities in the local area across the project.

Mr. Curtis Smith visited the site from 7-14 March 2022, from 10–17 June 2022, from 9-19 August 2022, and from 30 October–4 November 2022. The site visits included briefings from KCSA and Ivanhoe Mine Ltd., geology and exploration personnel, underground inspections of the Kakula, Kansoko and Kamoa 1 mines, site inspections of the Kamoa 1 decline portal and box cut and the sites for future mining, plant and infrastructure, discussions with other QPs and review of the existing infrastructure and facilities in the local area across the project.

Mr Jeremy Witley visited the Project from 15–22 August 2022. Mr. Witley went underground at both the Kakula and Kansoko mines, inspected drill core, observed active drill rigs in the field, inspected the on-site sample preparation facility, and observed the sampling methodology and security measures in place. The site visits also included discussions of geology and mineralisation interpretations with Ivanhoe's staff, focussing on deposit strike, dip, and faulting geometries.

Mr. William Joughin visited the site from 10–13 July 2017, 13–16 August 2018, 3–11 March 2022, and 31 October – 4 November 2022 to check geotechnical logging and inspect the ground conditions and support in the box-cuts, declines and areas with active mining and development at Kakula and Kansoko. The visits also included discussions with the on-site geotechnical team and a review of the geotechnical support being installed and the behaviour thereof over time.

Mr. Marius Phillips from DRA Global attended site from 12–14 December 2022. Site visit included concentrator visit, review of concentrator operating data and metallurgical accounting. Visited TSF, Laboratory and Smelter Site.

Mr. Alwyn Scholtz from DRA Global visited the site on several previous occasions with the latest being from 10 June – 18 June 2022. Briefings between site personnel, and other QP's visiting the site, was conducted, and he visited the Kakula mine underground, and surface operation, as well as Kamoa mine.

Mr. Andreas Savvas, Civil Engineer, visited Kamoa-Kakula TSF site 22–25 January 2019. His site visit included briefings with the geotechnical engineer undertaking the soils investigation of the TSF site, as well as inspections of the proposed TSF site.

2.5 Effective Dates

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

- Effective date of the Report: 6 March 2023.
- Date of the database closure Kamoa Mineral Resource estimate: 20 January 2020.
- Date of the database closure Kakula Mineral Resource estimate: 13 December 2022 (database closed for acceptance of new drillholes on 20 July 2022).
- The Kamoa Mineral Resource has an effective date of 30 January 2020.
- The Kakula Mineral Resource was depleted to account for annual production and has an effective date of 31 December 2022.
- Date of the Mineral Reserve estimate for Kamoa: 31 December 2022.
- Date of the Mineral Reserve estimate for Kakula: 31 December 2022.
- 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 IDP23, 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: Kamoa-Kakula 2023 PFS Section 19 Market Studies and Contracts, March 2023.
- 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: Kamoa-Kakula 2023 PFS, Section 4 Property Description and Location, March 2023.

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: Kamoa-Kakula 2023 PFS, Section 4 Property Description and Location, March 2023.

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: Kamoa-Kakula 2023 PFS, Section 4 Property Description and Location, March 2023.
- Kamoa Copper SA: Kamoa-Kakula 2023 PFS, Section 20 Environmental and Social, March 2023.
- 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, 28 January 2023.
- Kamoa Copper SA: Kamoa-Kakula 2023 PFS, Section 4 Property Description and Location, March 2023.

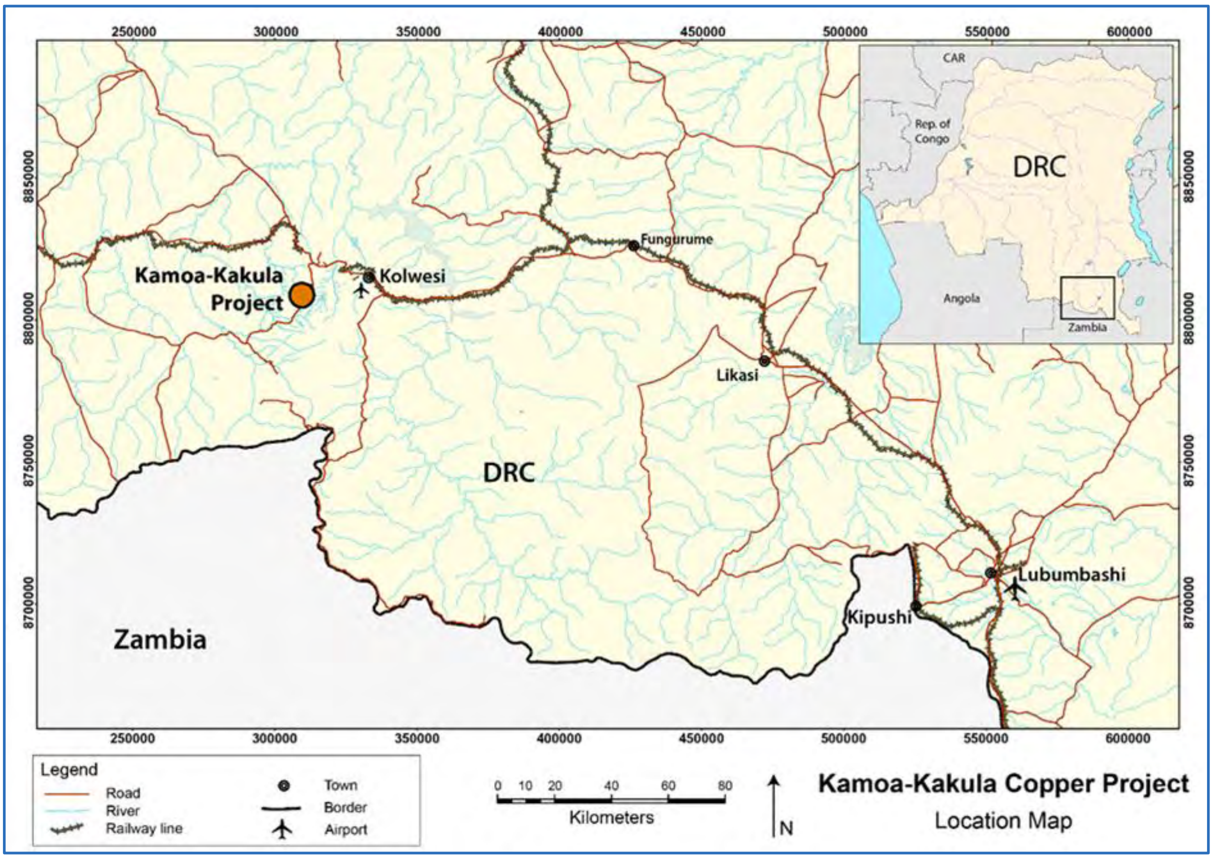
This information was used in Section 22 of the Report, and Section 14.12 for assessment of reasonable prospects of eventual economic extraction.

4 PROPERTY DESCRIPTION AND LOCATION

The Kamoa-Kakula Project is situated in the Mutshatsha territory in the 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



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$2.71 billion on 31 December 2022), 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 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:

1. Enter within the exploitation perimeter to proceed with mining operations.
2. Build the facilities and infrastructure required for mining exploitation.

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

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

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

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

4.2.4 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.

4.2.5 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.

Kamoa Copper SA updated ESIS was submitted in April 2022. The related environmental certificate is dated 13 July 2022. Kamoa Copper SA is in the process of submitting a new updated ESIS to the authorities.

Kamoa Copper SA notably obtained in 2020 exploitation permits for its classified facilities and applied in 2022 for updated exploitation permits covering new classified facilities, and is in the process of addressing the request from clarifications received from the relevant administration. In the meantime, with regard to DRC's expectations and in order to mitigate risks, Kamoa Copper applied for clearing authorisations (permis de déboisement) and paid, under duress, the related taxes as well as those related to classified facilities, required by the relevant administrations, while they were, in Kamoa Copper SA's view, legally challengeable. Kamoa Copper SA is actively following its 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 two year environmental audit concerning Exploitation Permits No. 12873, 13025 and 13026 was thus filed on 31 May 2022.

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.

4.2.6 Royalties

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

Pursuant to the 2018 DRC 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 3.5% for non-ferrous and/or base metals including copper.

4.3 Mineral Tenure

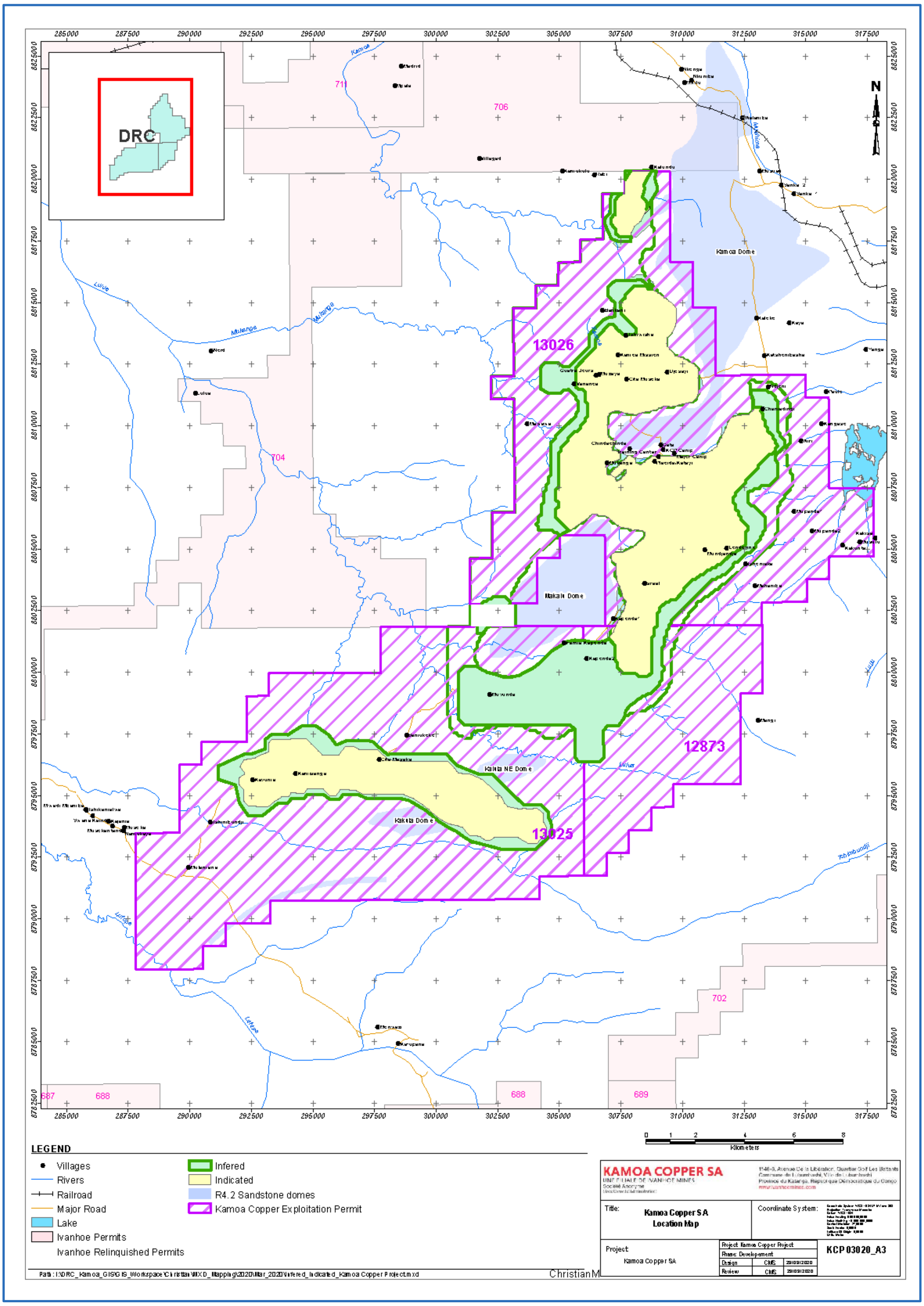
The Kamoa-Kakula Project consists of the Kamoa exploitation permits (Exploitation Permits No. 12873, 13025 and 13026 which cover an area of approximately 39,316 hectares). A mineral tenure summary table is provided in Table 4.1 and the mineral tenure locations are as indicated in Figure 4.2. The exploitation 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
				Subtotal	39,315.61

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

Figure 4.2 Project Tenure Plan



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 2021, and 2022, 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 4.2.4, 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 relocated 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.

As well as the north-south rail corridor, there is also a historical rail line connecting Kolwezi and the major DRC mining hub with the border town of Dilolo, approximately 420 km to the west, and a further 1,290 km from the DRC-Angola border to the port of Lobito, Angola on the Atlantic Ocean. The rail line passes near the Kamoa-Kakula exploitation permits. While the DRC portion of the line is in dilapidated condition, there are infrequent train services in operation. The Angolan side of the rail line was built by a Chinese consortium in 2014, and a 30-year concession was awarded in November 2022 to a consortium including Trafigura Pte Ltd, Mota-Engil Africa and Vecturis SA.

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 Workforce and Infrastructure

The workforce for the project is currently drawn from local villages, Kolwezi, Lubumbashi, other regions of the DRC, and internationally. The expatriate portion of the workforce is minimised and regulated. Transport is provided by KCSA from Kolwezi, and the local area, using buses and cars.

The existing infrastructure at Kamoa supports the current underground mining and processing operations for the Kakula, Kansoko Sud, and Kamoa 1 Mines, and the Phase 1 and 2 of the processing facility.

5.4 Power

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 Holdings Ltd.

Ivanhoe Mines Energy DRC recently (2021) completed the rehabilitation of six turbine generators at the Mwadingusha hydropower plant (HPP) in south east DRC and restored the plant to its installed capacity of 78 MW during the construction, and commissioning, of the first phase of the Kamoa-Kakula Concentrator. The securing of power for the Kamoa-Kakula Project is done by Ivanhoe Mines Energy DRC on a loan agreement from Kamoa with SNEL that will be repaid on a 40% discounted consumption charge.

For the Phase 3 upgrades, the Kamoa Board has agreed to extend the loan agreement with La Société Nationale d'Électricité (SNEL), for the upgrade of unit 5 (G25) at Inga II hydropower plant (HPP) in the South west DRC.

This unit will be capable to export 178 MW over a 1,700 km, 500 kV High Voltage Direct Current (HVDC) line to the Kolwezi area, where it is converted back into Alternating Current (AC) and tied into the 220 kV grid at the 220 kV SCK substation in Kolwezi. The SCK substation is a major 220 kV transmission station in the SNEL's southern network. Upgrading of the filter banks at Inga II, as well as at SCK, will also be part of the loan agreement, in order to boost the HVDC line's transmission capacity. Part of this project include the installation reactive power compensation equipment at SCK 220 kV substation.

The upgrading is part of a programme planned, to eventually overhaul, and power boost output. On completion of the upgrading programme, a combined total of 278 MW of long-term, clean electricity will be produced for the DRC's national grid.

However, to meet the power requirements of the Phase 4 expansion, a total of 440 MW supply will be required. Refurbishing costs have been included in the study to refurbish another INGA plant turbine that will supply the Project with additional 178 MW over the same lines as described above, this will increase supply to 456 MW.

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 smelter 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 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). PEA on the Kamoa and Kakula deposits was also prepared in 2020 as part of a Kamoa-Kakula Integrated Development Plan 2020 (Peters et al., 2020).

The Kansoko Mine has a Mineral Reserve that was previously stated in the Kamoa 2016 Pre-feasibility 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 was increased from 4.0 Mtpa to 6.0 Mtpa, and the total combined production rate was 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, and Kakula West Mineral Resource updates, and the Kamoa-Kakula 2019 PEA considering an 18 Mtpa plant expansion.

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

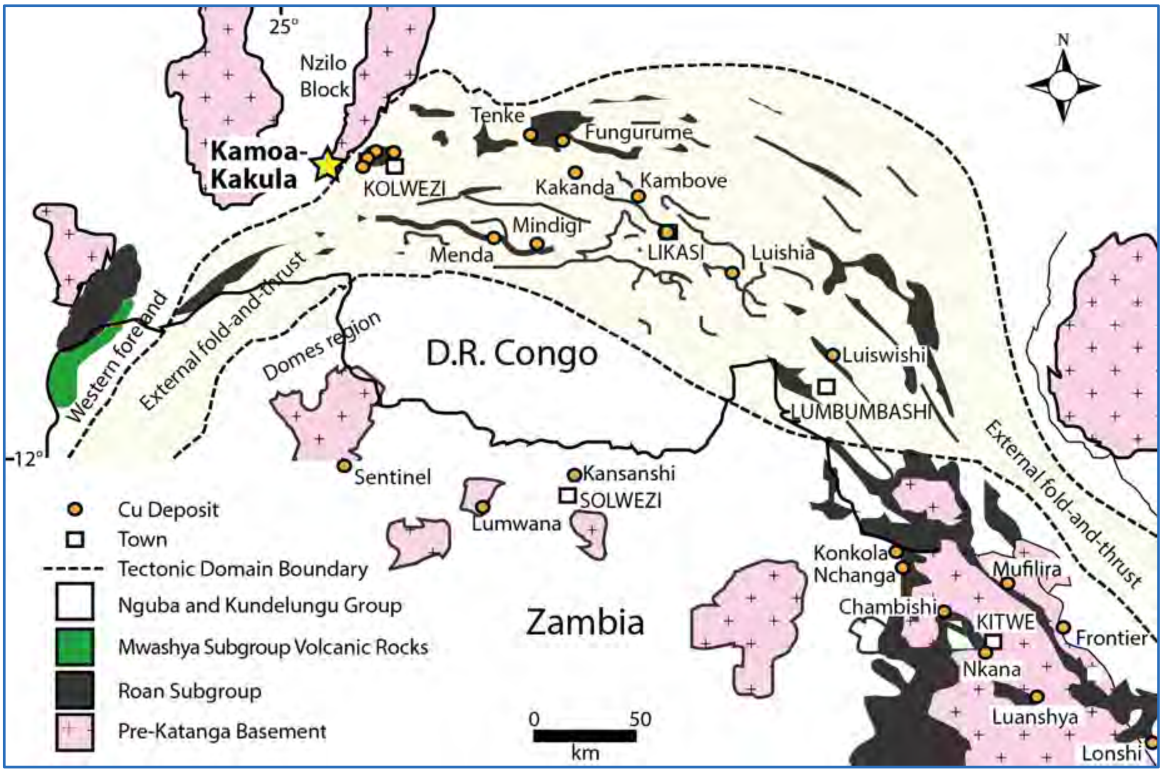
The previous Technical Report was the Kamoa-Kakula Integrated Development Plan 2020 with an effective date in October 2020. This included Mineral Reserves for Kamoa and Kakula, and the Kamoa-Kakula 2020 PEA considering a 19 Mtpa plant expansion.

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

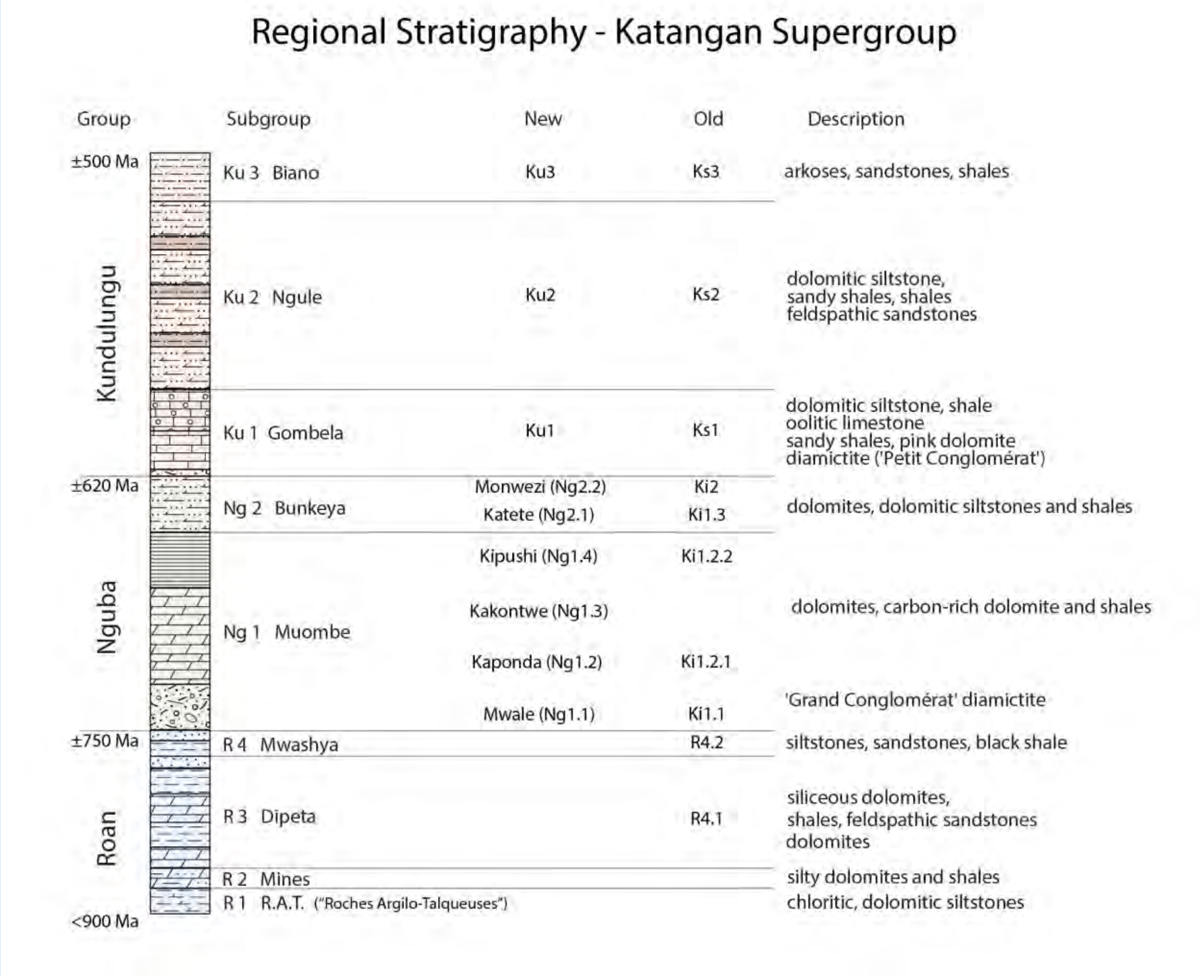


Source: Adapted from Schmandt et al (2013).

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. Some older images in this report may still use the earlier nomenclature.

Figure 7.2 Stratigraphic Sequence, Katangan Copperbelt



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

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 (Ng1.1).

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 siltstone 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).

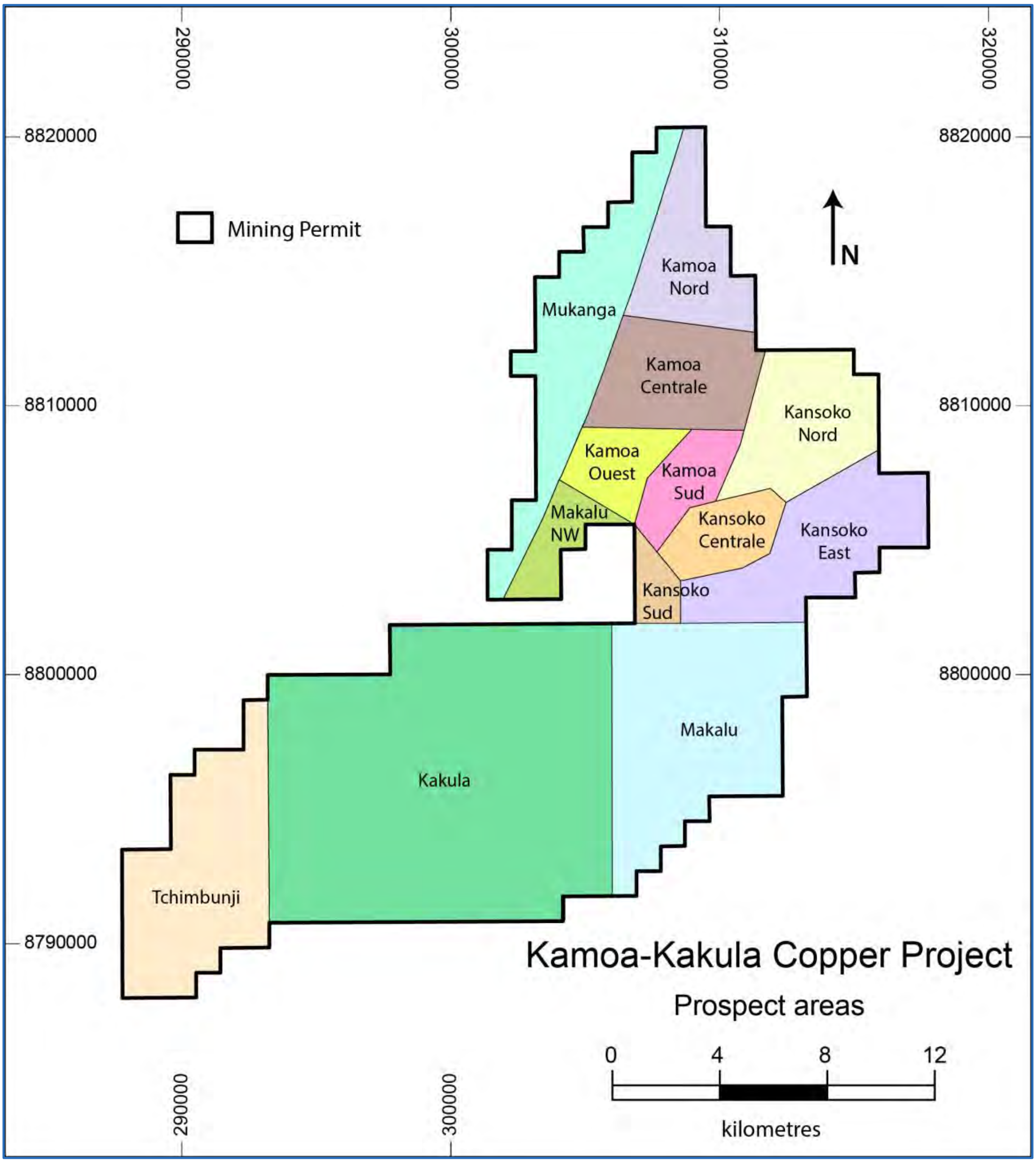
Andesite/dolerite sills occur as one or more, 5–80 m thick, apparently concordant tabular bodies in the extreme north-east of the Project area. The Katangan rocks in the Project area are weakly metamorphosed to lower greenschist facies. Alteration minerals include carbonate, chlorite, sericite, potassium 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 (Cu2S), bornite (Cu5FeS4) and chalcopyrite (CuFeS2). 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



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 (Ng1.1.1.1), which is overlain by a clast-poor diamictite (Ng1.1.1.3). Mineralisation is typically concentrated along the basal contact of this clast-poor diamictite, or in a locally-developed intermediate siltstone (Ng1.1.1.2) that separates the two diamictite units. The Ng1.1.1.2 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 (Ng1.1.2), 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 (Ng1.1.1.3) diamictite is absent.

The KPS is overlain by a thick sequence of diamictite with laterally discontinuous siltstone layers (Ng1.1.3). The Ng1.1.4 is a regionally developed bedded to laminated pyritic siltstone with intercalations of sandstone and minor gritty pebbles. The Ng1.1.4 is overlain by a thick (>300 m) unit of clast-poor diamictite (Ng1.1.5). A relatively thick (average 24 m), distinctive, cross-bedded sandstone separates the Ki1.1.5 from the overlying Ng1.1.6 diamictite, which is similar in character to the Ng1.1.5.

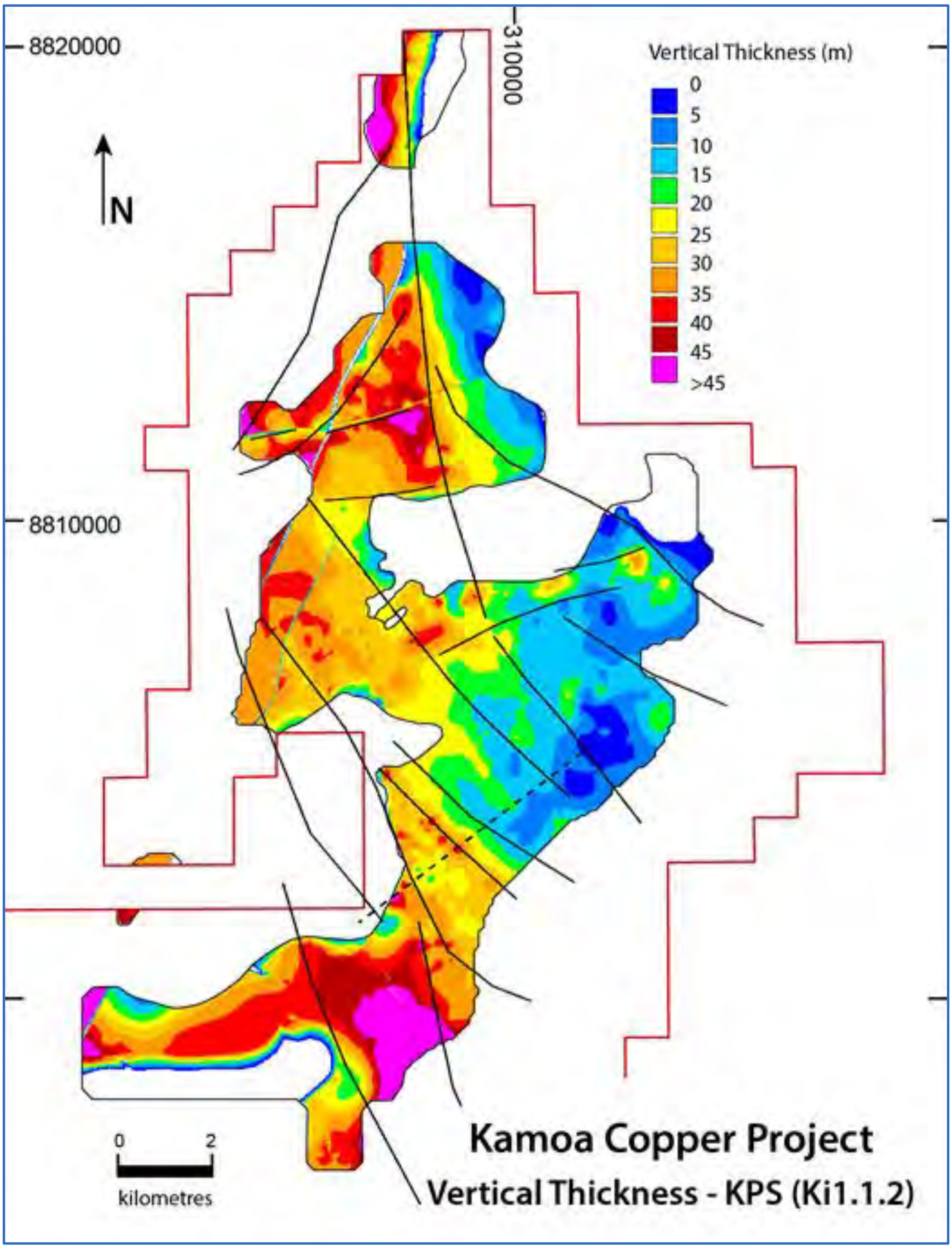
The stratigraphic units generally dip gently at 5–20° away from the Kamoa, and Makalu dome edges. The Kamoa, and Kamoa North areas, are particularly gently-dipping; Kansoko Sud, and Kansoko Centrale, generally dip at 10–20° to the south-east, with occasional steepening up to 30°. The steepest-dipping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

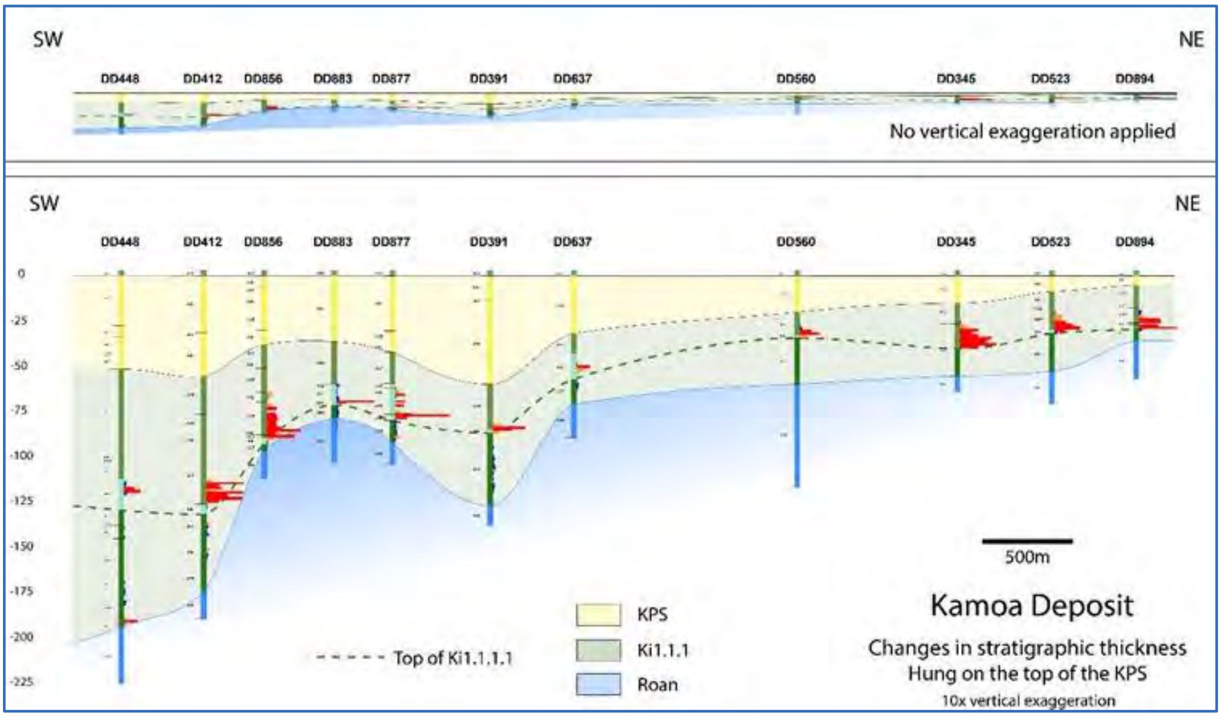
In the south-west, the thickening of the diamictite units is also marked by the development of thicker siltstone-sandstone-siltstone units, or the development of numerous siltstone units, comparable to the numerous siltstone units identified within the Ng1.1.1 at Kakula.

Figure 7.4 KPS (Ng1.1.2) Vertical Thickness



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)



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

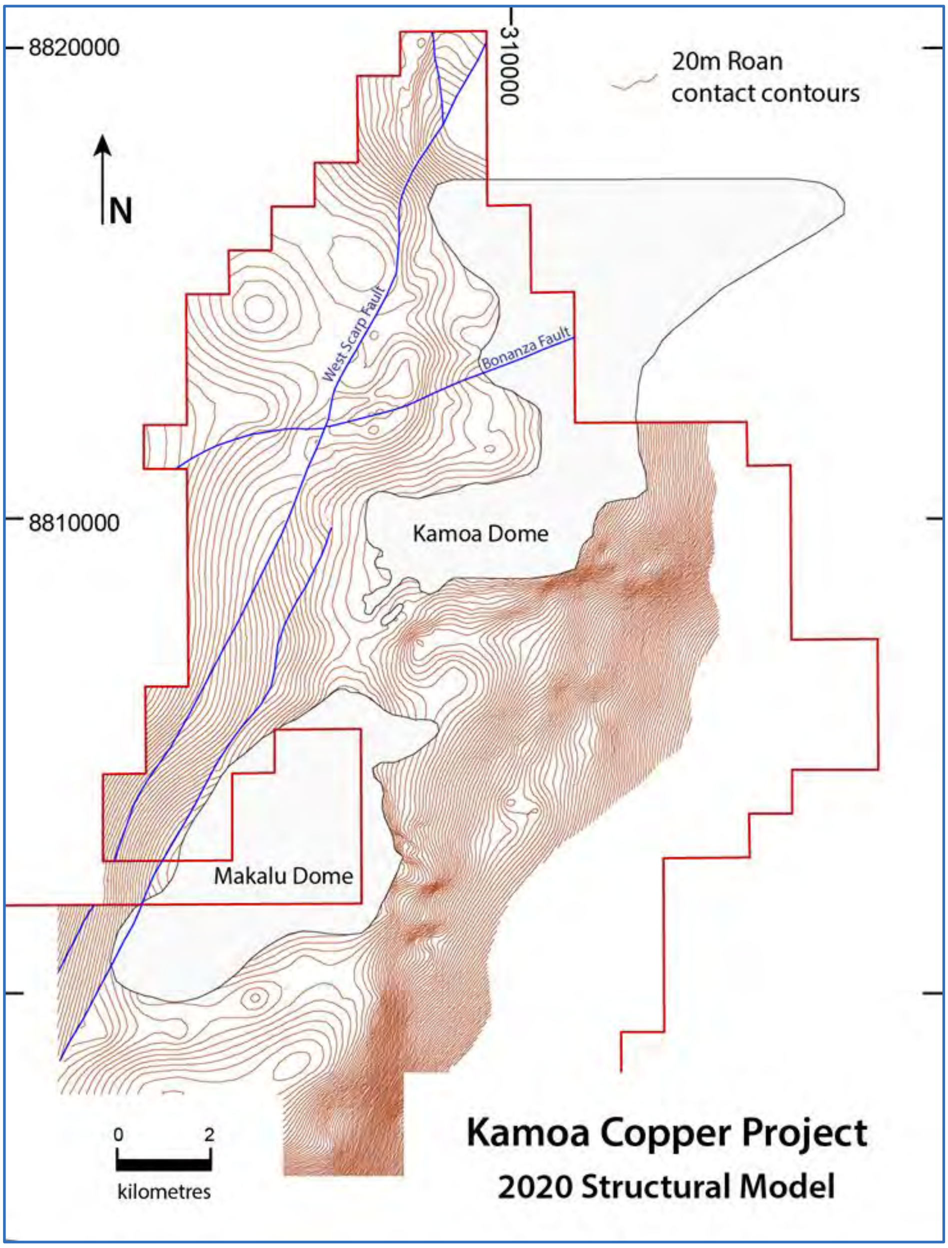
7.3.3 Structure

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–north-east 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 Mode and Contours (masl) for the Roan-Ng.1.1 Contact at the Kamoa Deposit



Ivanhoe, 2020.

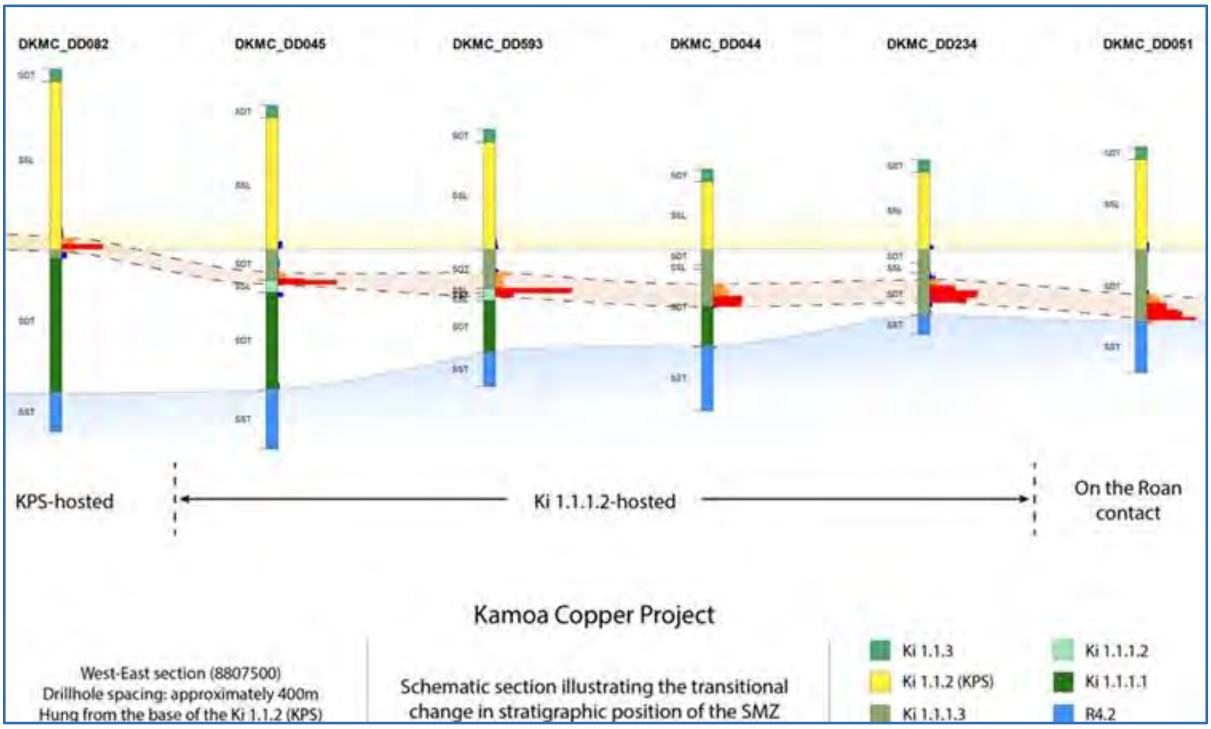
7.3.4 Mineralisation

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 (Ng1.1.1.1) is considered to be only weakly reducing, and thus generally hosts only low-grade (<0.5% TCu) mineralisation. The intermediate siltstone (Ng1.1.1.2) and clast-poor diamictite (Ng1.1.1.3) 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).

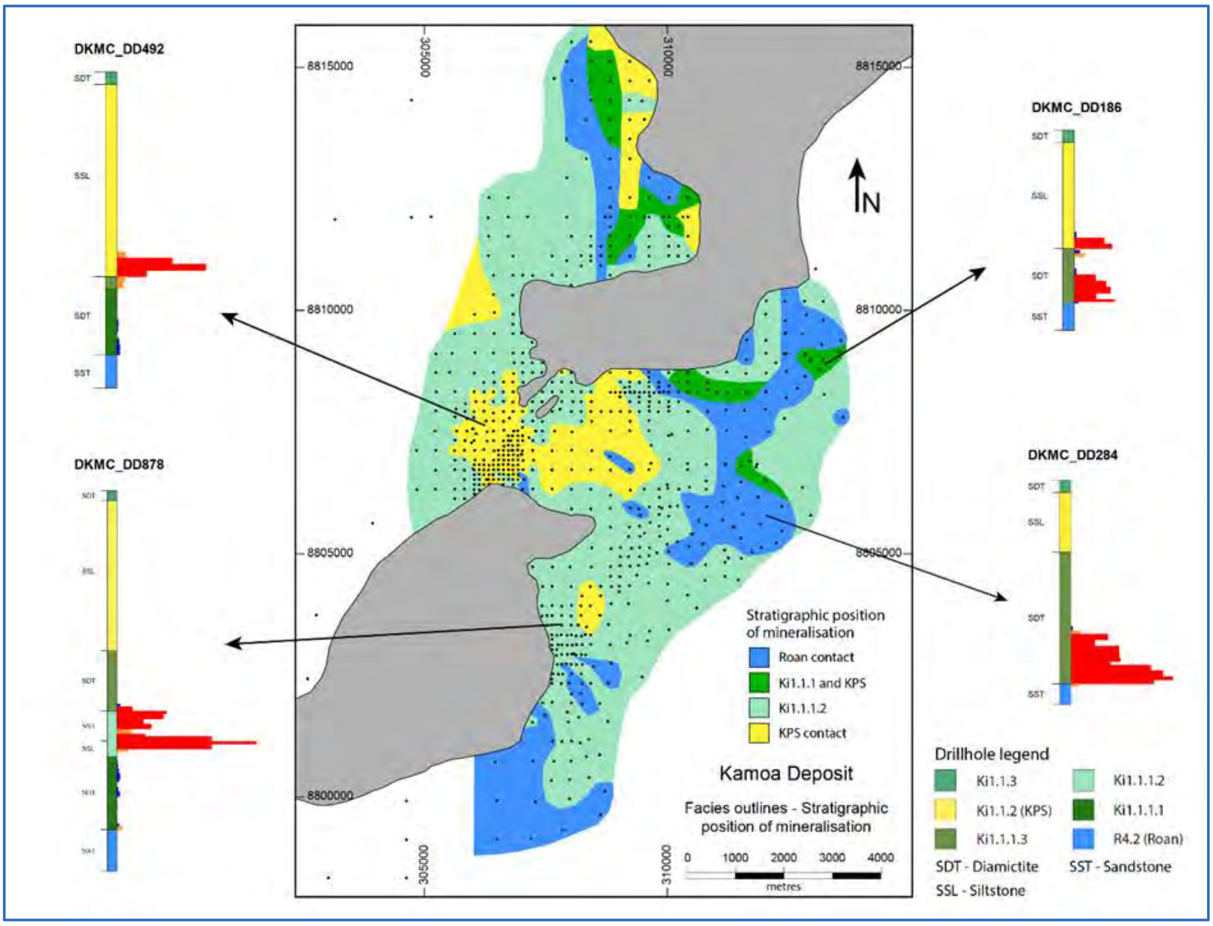
Figure 7.7 Stratigraphic Section Showing Continuity of Mineralisation Near Base of Ng 1.1.1.3 at the Kamoa Deposit (8807500N looking North)



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 (Ng1.1.1.3), 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



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

Continuity of higher-grade zones within these deposits is controlled by the local sub-basin architecture. Favourable sub-basin architecture, such as that at Kakula, can produce very strong continuity in excess of 4 km in both the thickness of the host siltstone, and occurrences of elevated grade. Where controlling factors are more juxtaposed, mosaic-patterns in terms of grade and thickness can form in the order of a kilometre in extent. At their edges, there can be significant changes to grade or thickness over a few hundred metres.

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. Where 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 (Ng1.1.1.2).

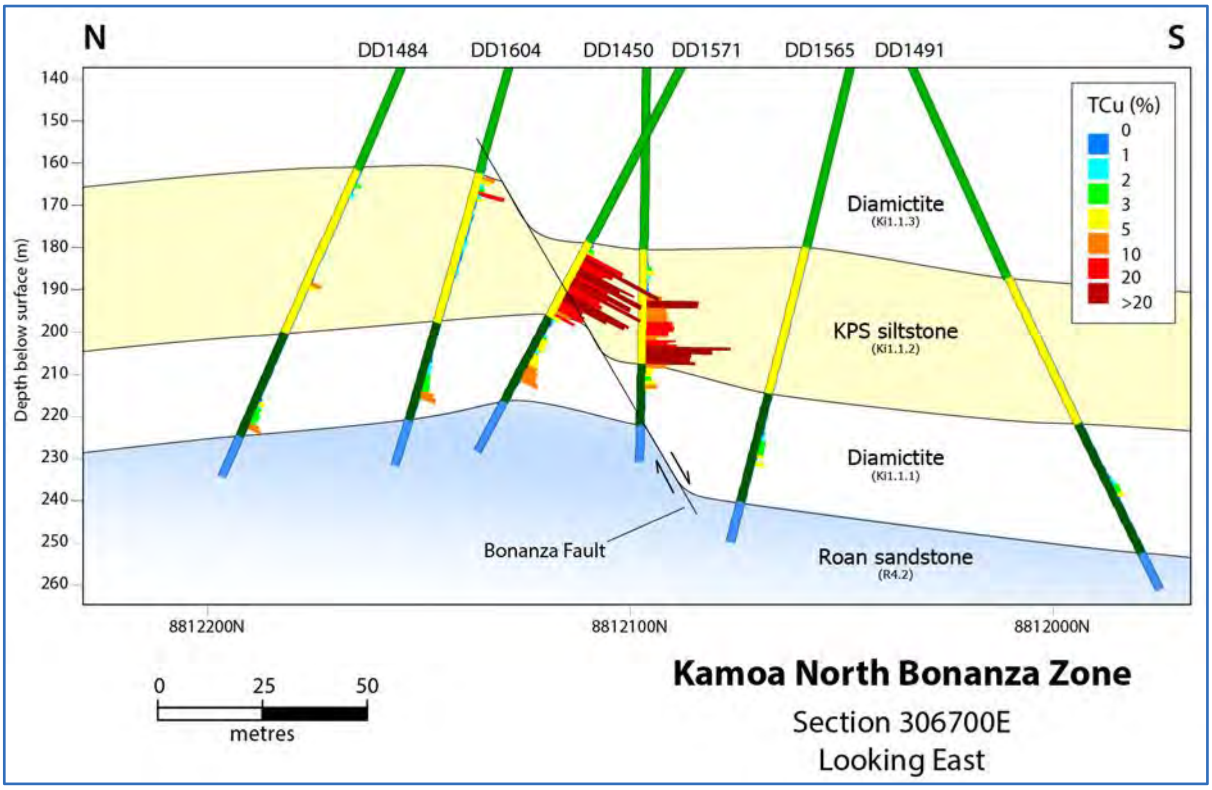
Supergene zones, in close proximity to dome edges, are typically fine-grained chalcocite dominant with secondary native copper and cuprite. The supergene zone may locally extend to depths of 250 m or more along fracture zones.

Since 2018, exploration has primarily focussed 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 (Ng1.1.2; refer to Figure 7.9). This has resulted in a stacked mineralised horizon, with the upper mineralised horizon of limited lateral extent but at a very high-grades hosted in the KPS (found in the vicinity of hole DD1450) and a lower horizon with typical diamictite-hosted mineralisation with extensive lateral continuity.

Drill sections 50 m apart 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 mineralisation 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



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 Ng1.1.1 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 Ng1.1.1 are generally clast-poor and are typically silt-rich. Numerous siltstones are developed within the Ng1.1.1, 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 (Ng1.1.1.2) 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 Ng1.1.1 and the development of high-grade mineralisation.

The shallowest portion of the Kakula deposit lies between the Kakula, and Kakula north-east 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.

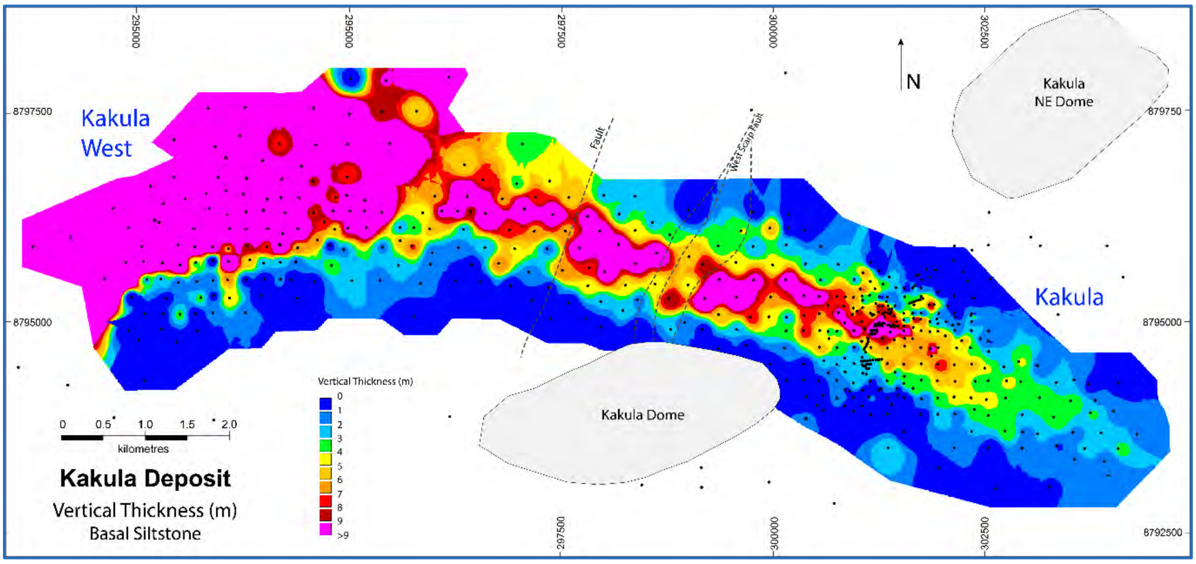
7.4.2 Thickness of Modelled Units

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 Ng1.1.1 generally thickens to the west. The Ng1.1.1 is considerably thicker than at Kamoa, with vertical thicknesses varying from 180 m to over 400 m at Kakula West. Locally, the KPS has been entirely eroded where it crops out along the domes, and the thickness for the Ng1.1.1 could not be estimated. A distinct zone of thickening within the KPS trends WNW-ESE across the eastern portions of the deposit, highlighting the rift controls during sedimentation; to the west, thickness patterns become more variable, reflecting the interaction of rift controls in different orientations (Figure 7.11). 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.

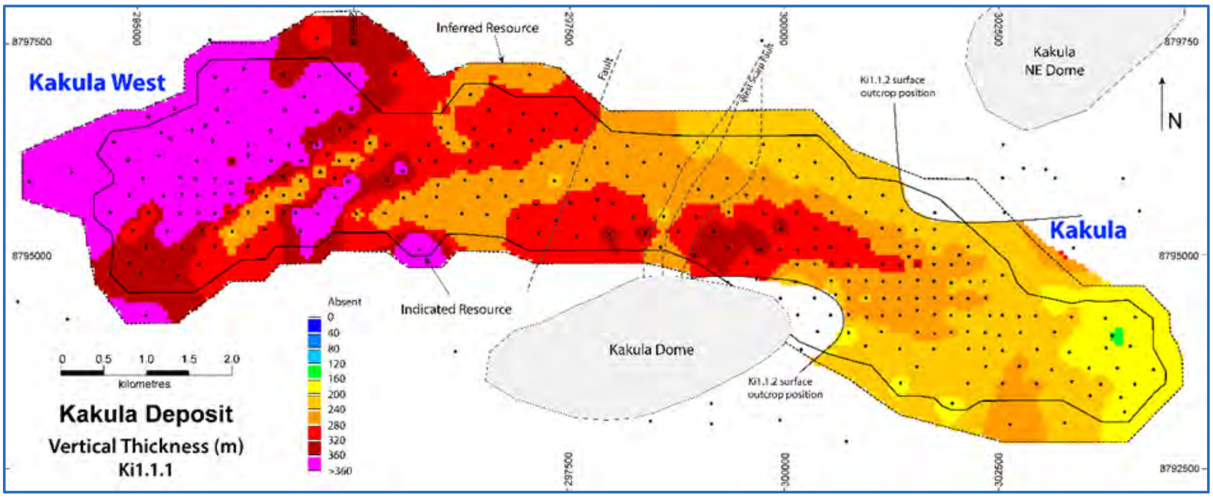
A pronounced east–north-east orientation in thickness trends is evident at Kakula West, and this observation has been incorporated into the search orientations used during grade estimation.

Figure 7.10 Vertical Thickness of the Basal Siltstone within the Ng1.1.1 at the Kakula Deposit



Ivanhoe, 2023. Vertical thickness estimated using an isotropic search.

Figure 7.11 Vertical Thickness of the Ng1.1.1 at the Kakula Deposit



Ivanhoe, 2023. Vertical thickness estimated using an isotropic search.

7.4.3 Structure

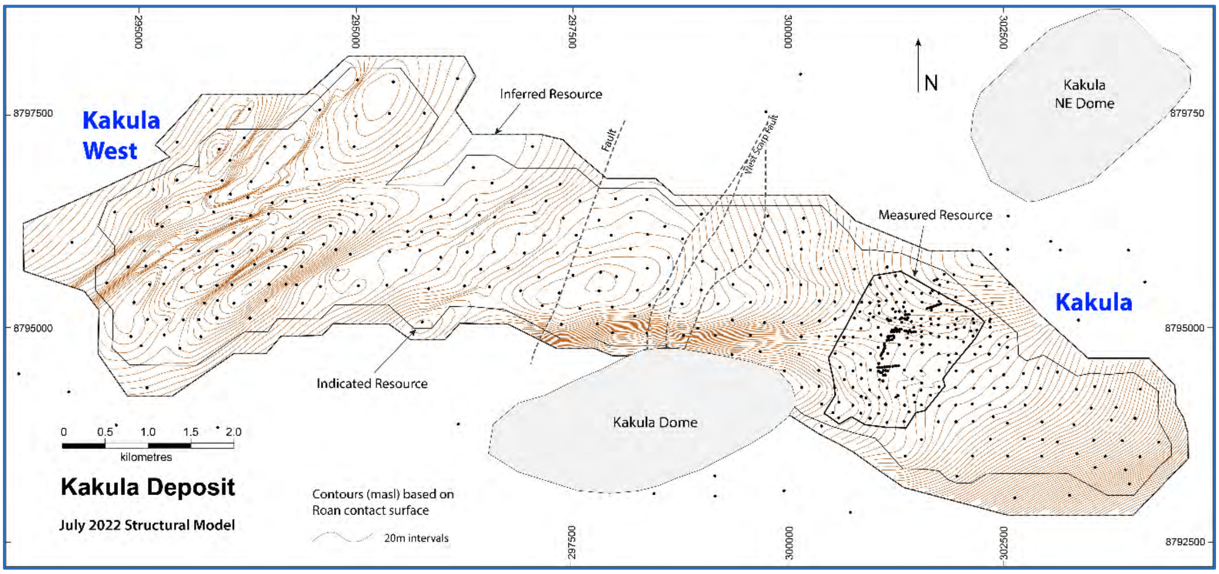
The geometry of the Kakula deposit is strongly influenced by extensional faults. Because the faults were active during deposition, a number of sub-basins were formed across the axis of a broad doubly-plunging antiform, and lithological units appear to drape across the extensional faults rather than having 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 Ng1.1.1–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 Ng1.1.2 is close to surface.

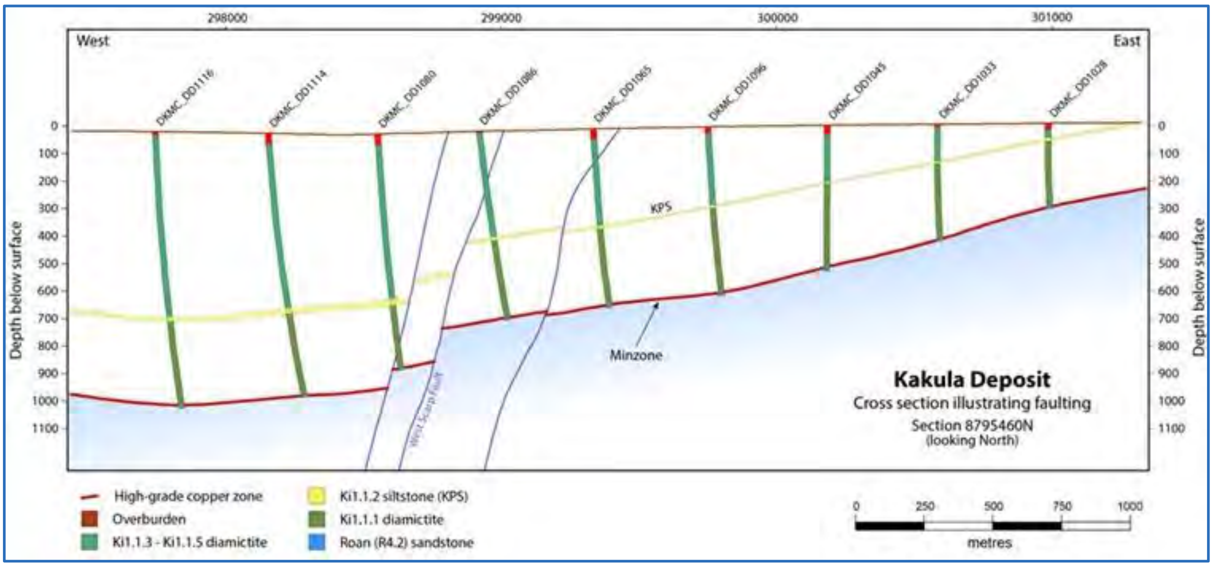
Younger brittle structures are also observed at Kakula that locally offset the mineralisation (Figure 7.12 and Figure 7.13). The most prominent of these faults trend north–north-east and are related to the West Scarp Fault. Additional observed structures in drill core include steeply-dipping chaotic breccias and gouges, and cohesive "crackle" breccias.

Figure 7.12 Structure Model for the Kakula Resource Area Showing Contours (masl) for the Ng1.1.1–R4.2 (Roan) Contact



Ivanhoe, 2023.

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



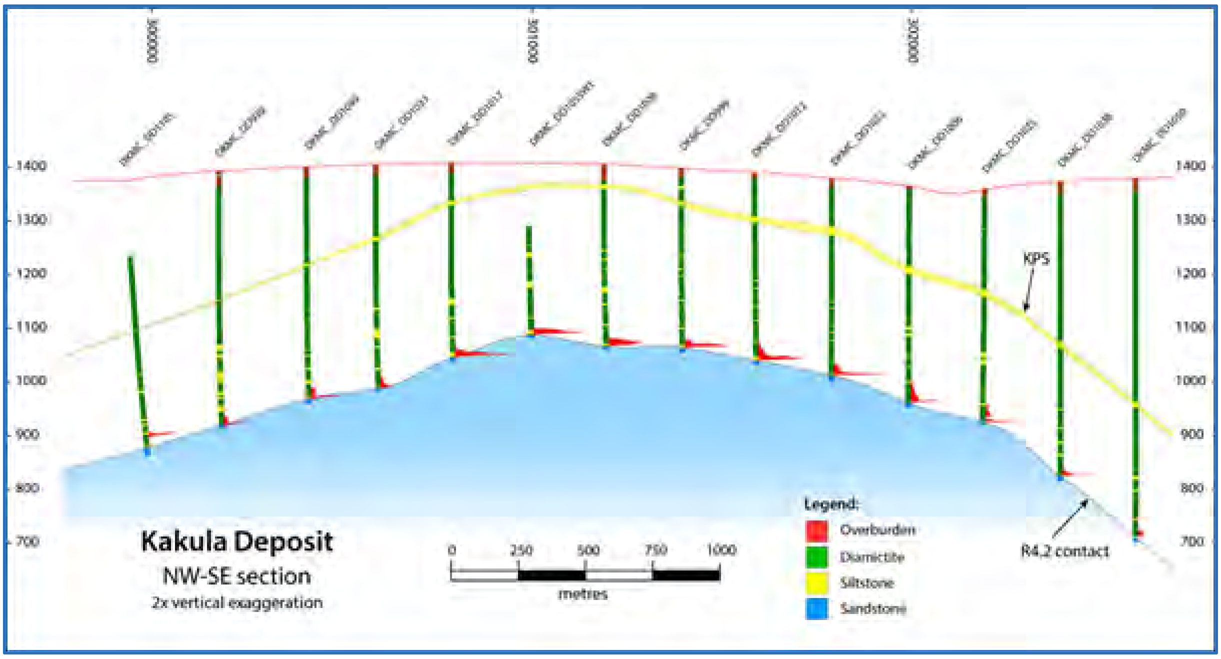
Ivanhoe, 2018.

7.4.4 Mineralisation

The Kakula deposit is currently delineated over an area of 14 km by 5 km. 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 south-east.

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.

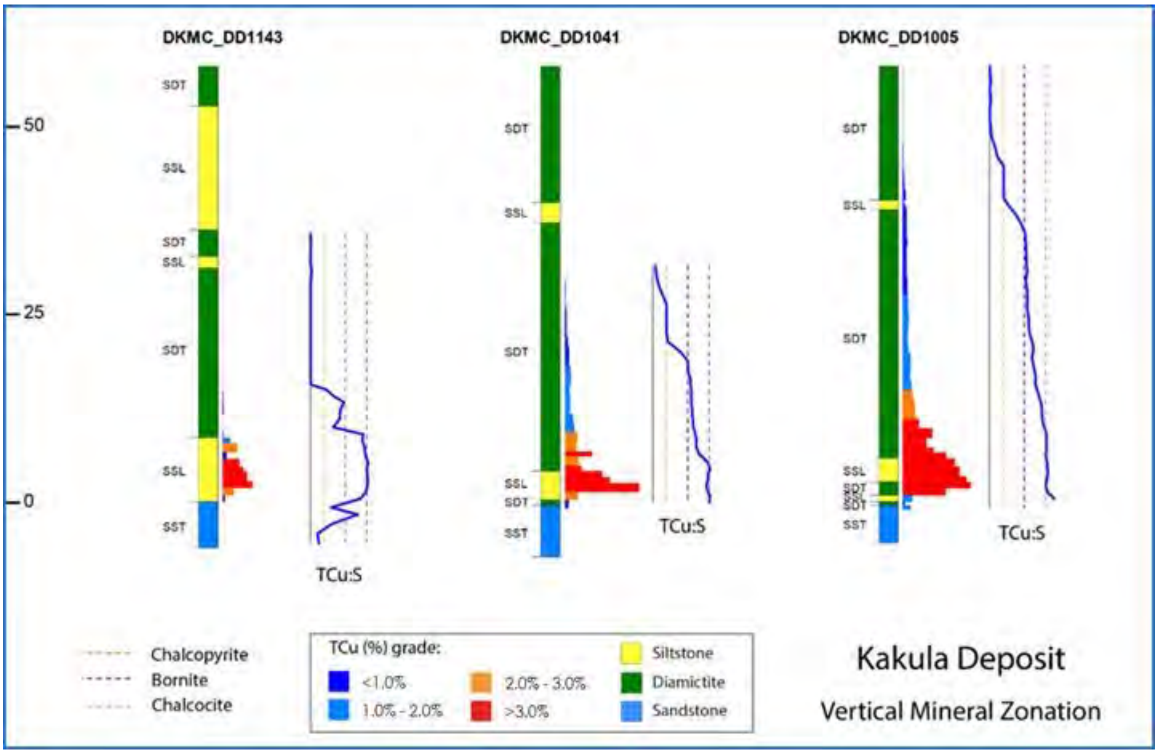
Figure 7.14 North-west to South-east Section through Kakula Illustrating the Numerous Siltstone Units Developed Towards the Base of the Ng1.1.1



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



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 north-west, the mineralisation turns to the west, with alignment along 105°. At Kakula West, well-developed growth faults control the alignment of thickness and grade trends that vary around north-easterly orientations. The orientations of the controlling growth fault features have 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 MSA 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 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 (Ng1.1.1.2) 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). Localised 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.
Down-hole 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.
Down-hole 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.
Airborne radiometrics and gravity	2022	Airborne radiometric and gravity surveys were completed over the Kakula West area as part of a broader airborne geophysical programme over the exploration licences held to the west of Kamoa-Kakula

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 K2O 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).

9.6 Exploration Potential

The Kamoa-Kakula Project area is underlain mainly by sub-cropping 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 thicknesses 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 south-eastern boundary of the high-grade trend within the Mineral Resource estimate area is defined solely by the current limit of drilling, and the deposit remains open in this direction.

9.7 Comments on Section 9

The MSA 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. 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 of 2 December 2022, there were 2,808 core holes 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 20 July 2022 for acceptance of new drillholes. The assay data available up until 13 December 2022 were included in the modelling. The December 2022 Kakula Mineral Resource estimate used 645 drillhole intercepts.

The 1,165 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.

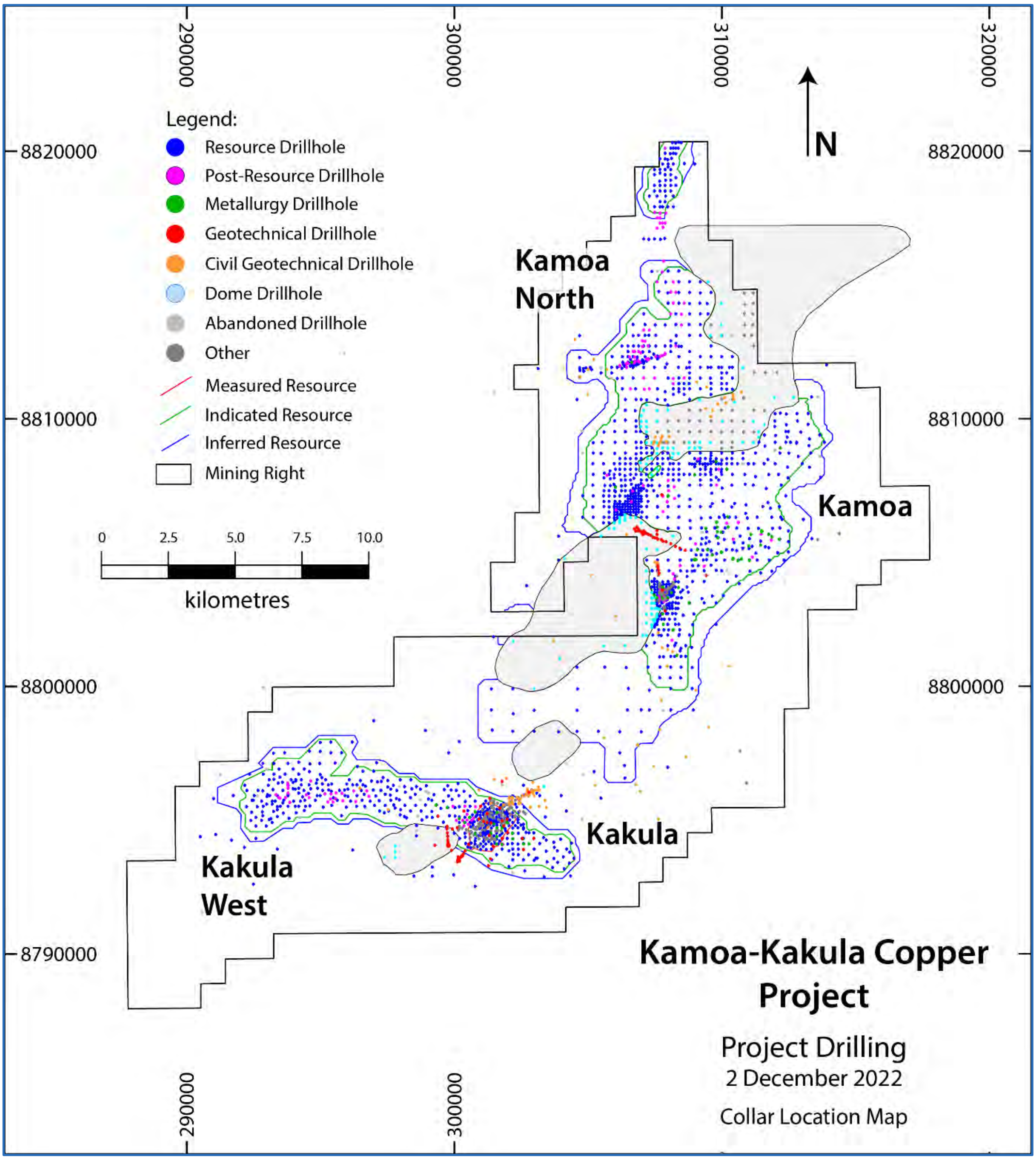
Core holes typically commence collecting cores at PQ size (85 mm), reducing to HQ size (63.5 mm), and where required by ground conditions, further reducing to NQ size (47.6 mm).

Table 10.1 Drilling Statistics per Drill Purpose for Core holes (as of 2 December 2022)

Drill Purpose	Count (Active)	Metres (m)
Kamoa estimate (Jan 2020)	998	288,140.7
Kamoa (post-estimation)	139	30,270.0
Kakula estimate (Dec 2022)	645	246,799.5
Kakula (post-estimation)	42	21,179.0
Surface Exploration	8	2,851.9
Underground Exploration	24	1,491.8
Domes	107	7,856.3
Metallurgy	132	13,996.2
Geotechnical	107	13,352.2
Civil Geotechnical	102	2,712.4
Condemnation	51	1,177.8
Cover Drilling	248	37,181.4
Hydrogeology	74	7,696.0
Abandoned	131	28,892.5
Total	2,808	703,597.5

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'. Surface 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



Ivanhoe, 2023. 'Other' includes exploration drillholes, condemnation drillholes, cover drillholes and hydrology 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.

Core holes 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 was photographed both dry and wet prior to sampling. All Kamoa core was subject to magnetic susceptibility measurements; these are being done selectively 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 at Kakula is generally good, with recovery data averaging 94% within the mineralised zone.

10.4 Collar Surveys

All drill sites were 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 were 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 were used.

10.4.2 Kakula

As of 20 July 2022, only three drillholes (DKKL_DD0048, DKKL_DD0083 and DKKL_DD0089) used in the Kakula Mineral Resource estimate lacked an independent collar survey. In these cases, the planned coordinate positions were used.

10.5 Down-hole Surveys

10.5.1 Kamoa

Core hole orientations ranged from azimuths of 0° to 360°, with down-hole 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. Down-hole surveys for most drillholes were performed by the drilling contractor at approximately 30 m intervals for 2009 drilling, and at a maximum interval of 50 m for 2010, through 2020 drillholes, using a Single Shot digital down-hole 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 core holes were not down-hole 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

Down-hole surveys for most drillholes were performed by the drilling contractor at approximately 3 m to 6 m intervals down-hole 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, dedicated geotechnical drillholes and underground mapping. 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 Section 16.1.

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 139 drillholes (30,270.0 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 ahead of mining in new production zones.

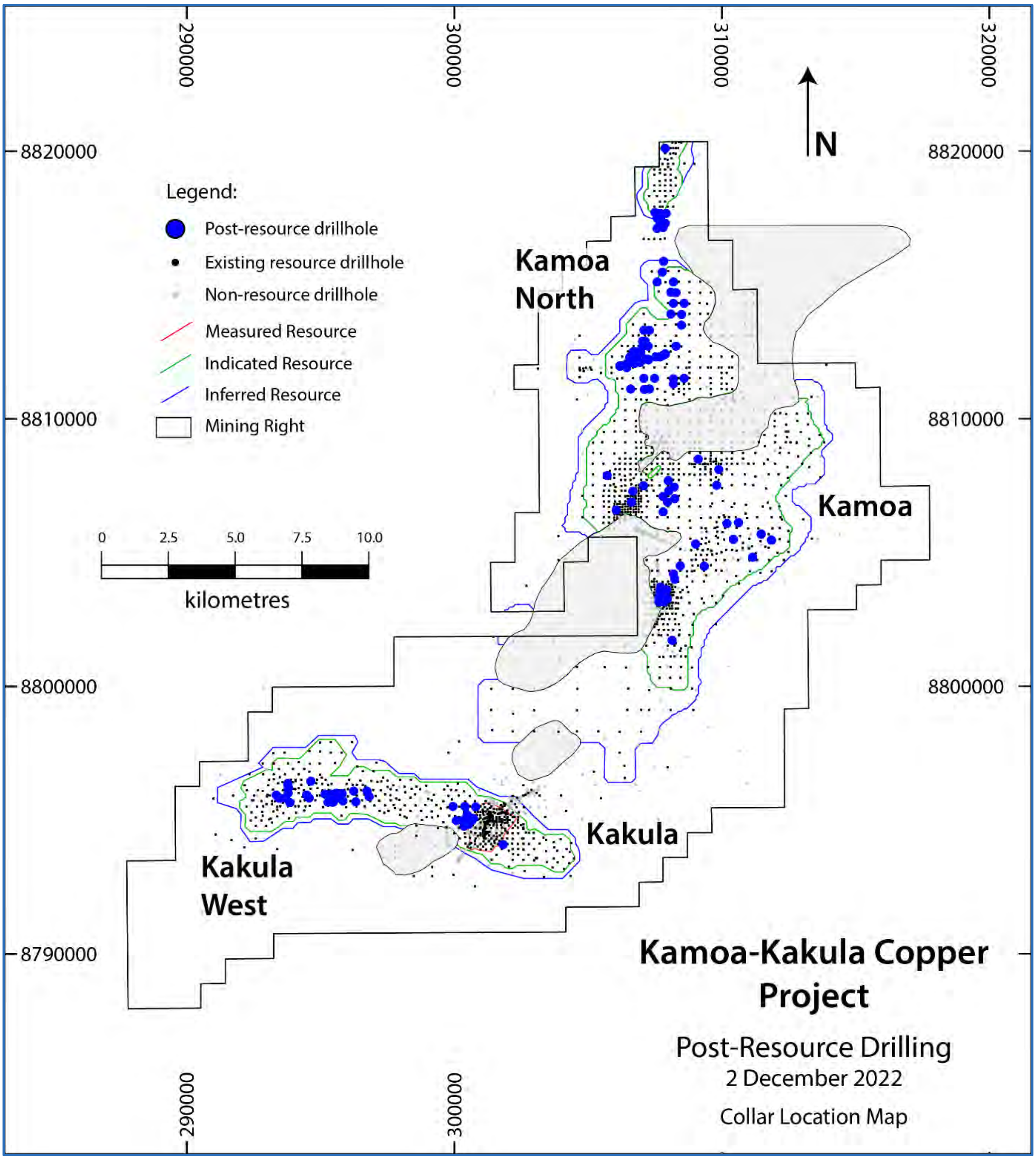
Although a few of the newer drillholes are very high-grade, and may change the grades locally, the majority of the holes are within the existing model, and the MSA QP considers that the new drilling should have no material effect on the overall tonnages and average grade of the current Mineral Resource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

10.8.2 Kakula

Since 20 July 2022, Ivanhoe completed an additional 42° core drillholes (21,179.0 m) at Kakula as grade control infill holes ahead of mining at Kakula, or as resource infill drillholes at Kakula West in support of future mine planning in this area. The collar locations of the core holes 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 MSA QP considers that this new drilling should have no material effect on the overall tonnages and average grade of the Indicated Mineral Resource.

Figure 10.2 Plan View Showing Kamoa-Kakula Drillholes Completed Since Construction of the Respective Mineral Resource Models (at 2 December 2022)



Ivanhoe, 2023.

10.9 Comments on Section 10

The quantity and quality of the lithological, geotechnical, collar, and down-hole survey data collected in the core drill programmes is sufficient to support Mineral Resource. The MSA 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.
- Down-hole surveys provide appropriate representation of the trajectories of the core holes.
- 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 programmes, 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 Ng1.1/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 (Ng1.1.2), and mineralised basal diamictite was conducted as follows:

- The mineralised zone was sampled on 1 m sample intervals (dependent on geological controls).
- The KPS (Ng1.1.2) 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 a rotating diamond saw blade. The cut line (for splitting) is typically offset from the core orientation line by 1 cm clockwise looking down-hole, 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 core hole 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 top size, 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.

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

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	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 ISO:17025	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 was crushed to nominal 2 mm using jaw crushers. A quarter split (500 g to 1,000 g) was pulverised to >90% -passing 75 µm, using the LM2 puck and bowl pulverisers. A 100 g split is sent for assay; three 50 g samples were kept as government witness samples, 30 g for Niton analysis, and approximately 80 g of pulp was retained as a reference sample. The remaining coarse reject material was 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 recording the weight prior to shipping. Certified reference materials (CRMs) and blanks were 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.

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 were used at Kakula. The year designations indicate the year the material for the blank was collected. A commercial low-grade CRM (OREAS22D) was 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. The current procedure is for blank samples to be 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. The current procedure is for blank samples to be placed after visually-observed 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), located in South Africa. To date, a total of 63 commercially available CRMs have been used at Kamoa, of these, 20 were 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 were inserted with a 5% insertion rate, and the CRM published value was matched to the expected mineralisation grades. CRMs were 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 used 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 programmes and were 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 down-hole 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 down-hole 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 back-up 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 were 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 MSA QP, the sampling methods are acceptable, consistent with industry-standard practice, and adequate for Mineral Resource 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 QP Verifications

Between 2009 and 2020, previous QPs conducted multiple site visits and reviews of the data available to support Mineral Resource estimation.

Reviews included checking of collar co-ordinates, drill collar elevations and orientations, down-hole and collar survey data, geological and mineralisation logging, assay, and specific gravity data. These reviews were documented and no significant errors were noted that could affect Mineral Resource estimation.

Since 2020, the MSA QP has conducted a site visit which included underground visits, reviews of drill core, sampling procedures, active surface drilling, geological logging and SG data collection. Close-spaced underground grade control sample and mapping data was also reviewed and compared to the Mineral Resource model. Monthly production data since April 2021 has been reconciled to the short-term and Mineral Resource model to ensure the geological modelling, domaining, and parameters used in the grade estimation are optimal based on all available data.

12.2 QA/QC Review

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

Previous QPs conducted periodic reviews of the QA/QC data between 2009 and 2013. Since 2013, the vast majority of QA/QC data have been reviewed by Mr. Dale Sketchley, P. Geo. of Acuity Geoscience Ltd, most recently in January 2023 (Acuity, 2023).

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 were 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.

12.3 Copper Grade Witness Sampling

Previous QPs conducted multiple phases of witness sampling between 2009 and 2017. With the underground exposure at both Kakula and Kansoko, detailed channel sampling and underground mapping, and production results since April 2021 confirm the estimated grades in the Mineral Resource model, it is no longer considered necessary to collect witness samples.

12.4 Comments of Section 12

The MSA 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. Underground exposure and monthly production results have served to further support the geological interpretations used and the range and variability of estimated grades.

Therefore, the collected data can support Mineral Resource 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 SA discovered the Kakula deposit which has significantly higher copper head grades compared to the Kamoa deposit. Based on this, the Kakula project was fast-tracked. Metallurgical testwork on the Kakula deposit was initiated in 2016, and conducted by Zijin Laboratories located in China, as well as XPS Laboratory located in Canada. Following the successful results obtained by both parties, further flow sheet development testwork was conducted at XPS during 2017–2018 as part of the Kakula Phase 1 pre-feasibility study phase. The Kakula Phase 1 and Phase 2 Concentrators have since been successfully commissioned in 2021/2022 and ramped up to design throughput.

For the purpose of the Kamoa-Kakula 2023 PFS, the metallurgical testwork conducted on the Kamoa deposit has been referenced in detail as background to the IFS4a flow sheet development process. A summary of the metallurgical testwork conducted on Kakula material for the Kakula Phase 1 pre-feasibility and feasibility studies is provided.

13.1.1 Metallurgical Testwork on the Kamoa Resource

A number of testwork phases were completed from 2010 to 2015 by various parties. These campaigns targeted a final product grade of 30% Cu, at a minimum recovery of 85% Cu, while maintaining SiO_2 levels below 14%.

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 Kamoa 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 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 IFS4a 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 IFS4a 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 IFS4a 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 IFS4a 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 IFS4a, 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 IFS4a 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 Pre-feasibility 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 to both.

13.2 Metallurgical Testwork on the Kamoa Resource

A number of testwork phases were completed from 2010 to 2015 by various parties. These campaigns targeted a final product grade of 30% Cu, at a minimum recovery of 85% Cu, while maintaining SiO_2 levels below 14%.

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 Kamoa 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

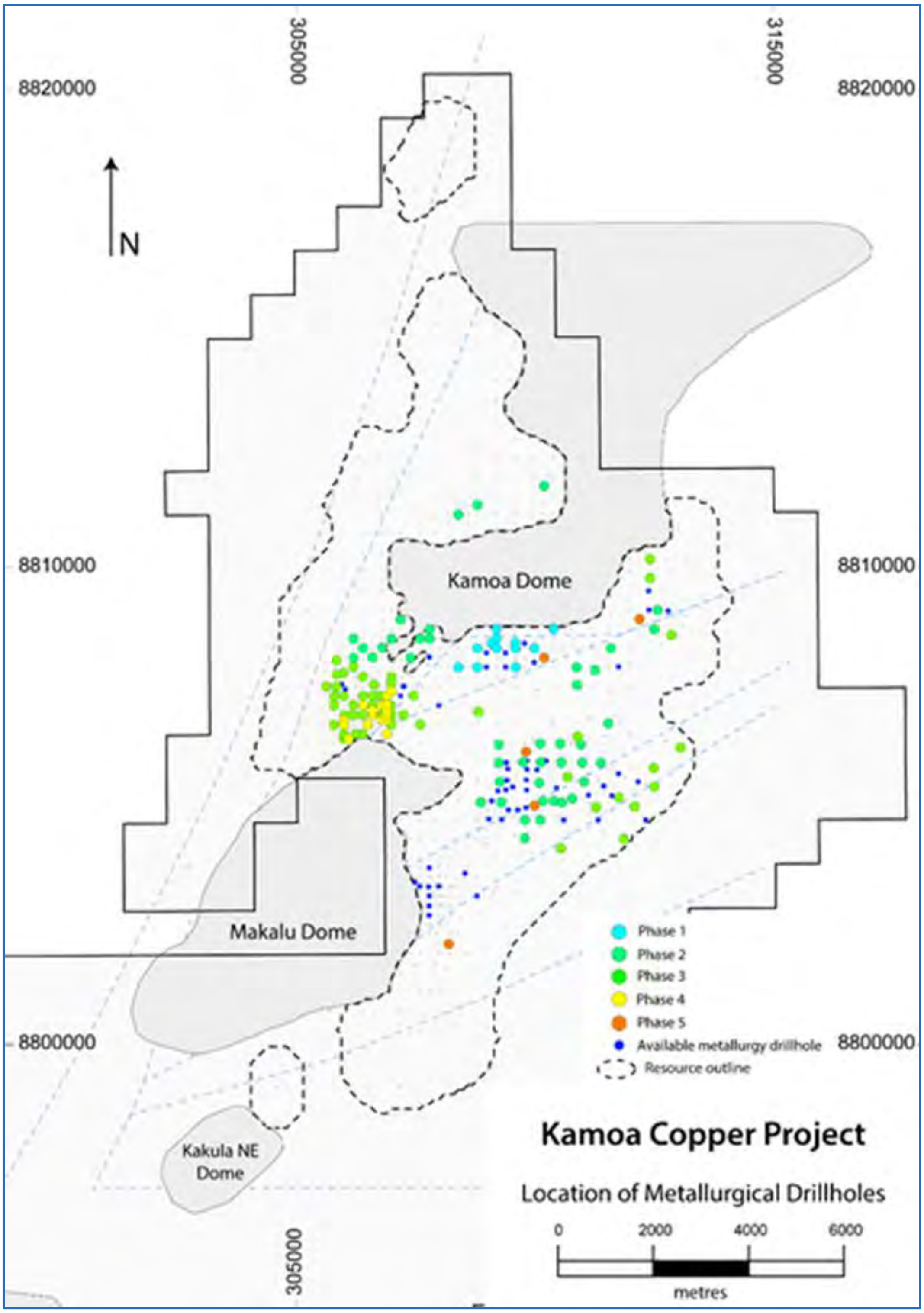
Phase	Study	Comminution	Flotation	Mineralogy	Period	Comment
1	Concept	Mintek	Mintek	SGS Johannesburg	2010–2011	Grab Samples
2	SS/PEA	Mintek	Mintek/XPS	XPS	2011–2012	Representative Composites
3	SS		XPS	XPS	2012–2013	Composites
4	SS	Mintek	XPS	XPS	2013	Open Pit
5	SS/PFS	Mintek	XPS/Mintek	XPS	2013–2014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



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

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

13.2.3.1 Competence (SMC Test) Summary

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

Table 13.3 SMC Test Results as Axb Value Range

Phase	Diamictites (Hypogene, Supergene and unmineralised)	Oxide	Pyritic Siltstone (mineralised and unmineralised, 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 above.

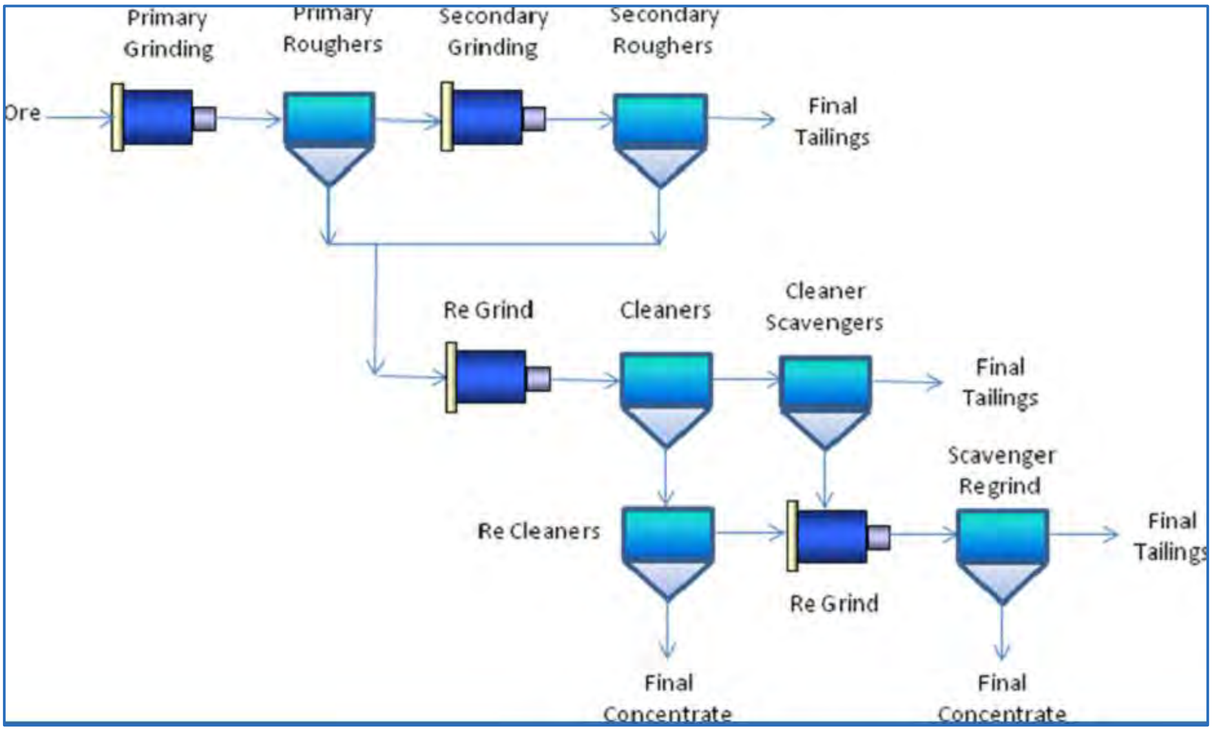
The samples were subjected to some basic bench scale testing including grinding, rougher flotation, concentrate, and tailings regrind and cleaner flotation optimisation. The separation work was supported by chemical and mineralogical analyses.

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



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.

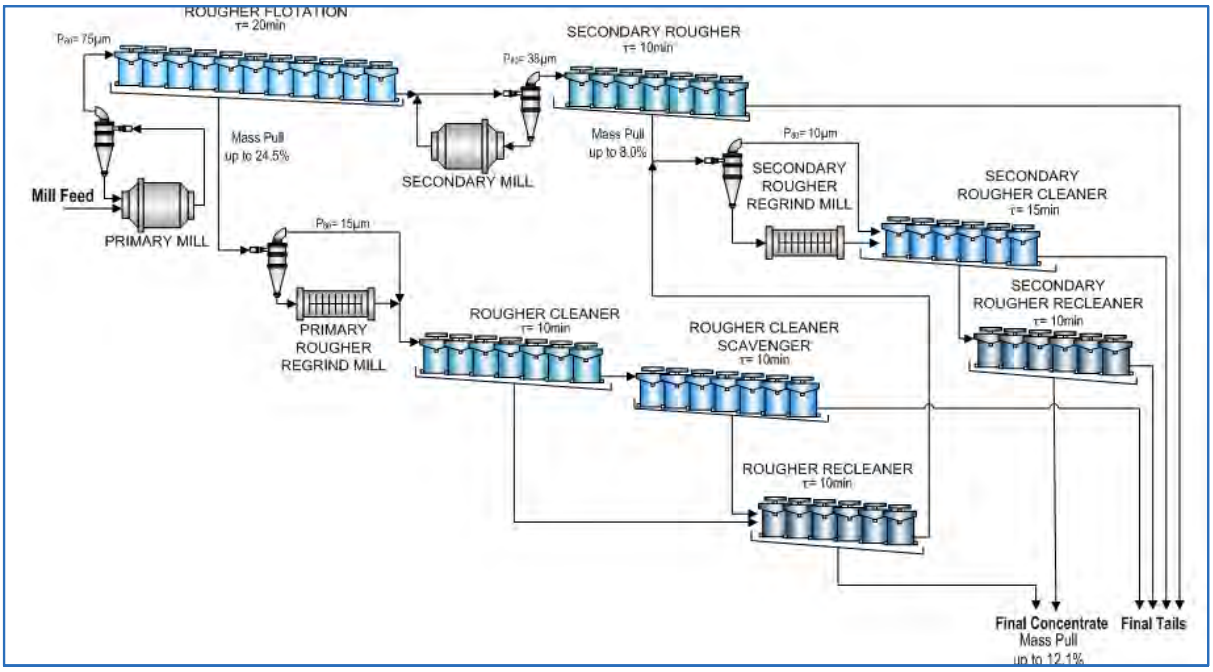
A testwork programme was performed on drill core samples from all major areas of the expanded resource, namely, Kamoa Sud, Kansoko Sud, Kansoko Centrale and Kansoko Nord. Samples were also taken from Kamoa Ouest; however, this area did not form part of the Kamoa 2017 PFS mine plan. Composites from the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



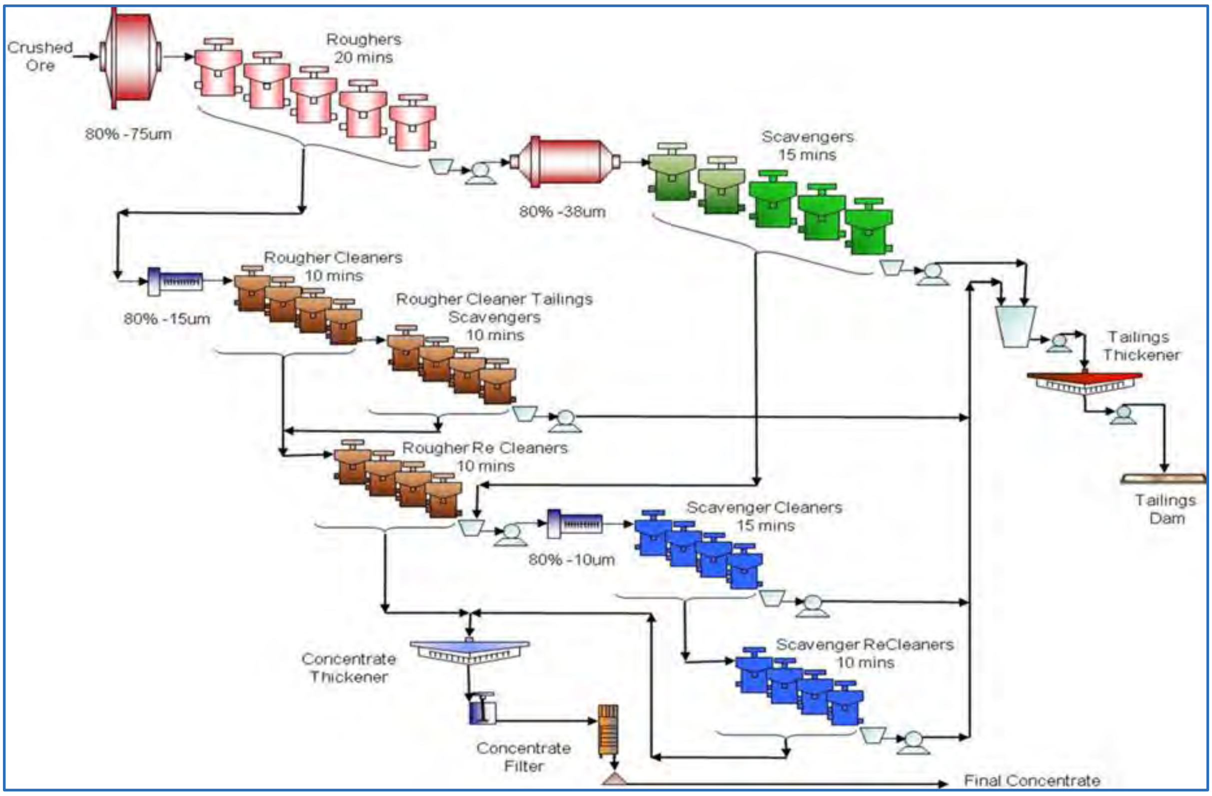
Hatch, 2013.

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



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 regrind 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 regrind 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 regrinds, and selective cleaning tests.

- The coarsening of the P_{80} of the primary, and secondary, regrind mill products resulted in low Cu grades and high SiO_2 content in the final concentrate. This confirmed that the optimum grind for the regrind circuit was P_{80} of 15 µm, and 10 µm for primary, and secondary regrind 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.

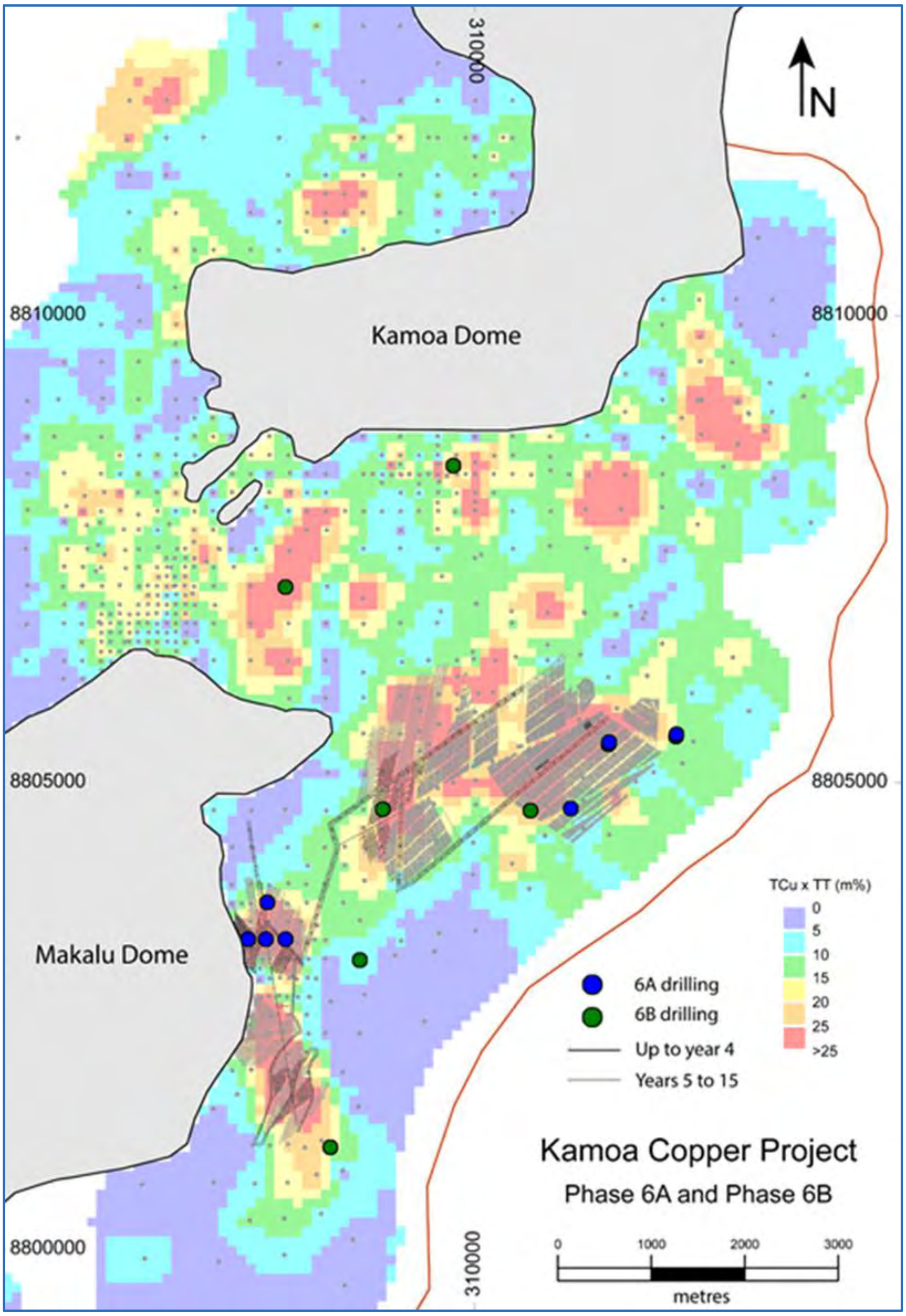
13.2.5.1 Phase 6 Comminution Testwork – Mintek

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



Ivanhoe, 2016.

Table 13.10 Phase 6 Comminution Summary

Sample ID	SG	BRWi	BRWi (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.

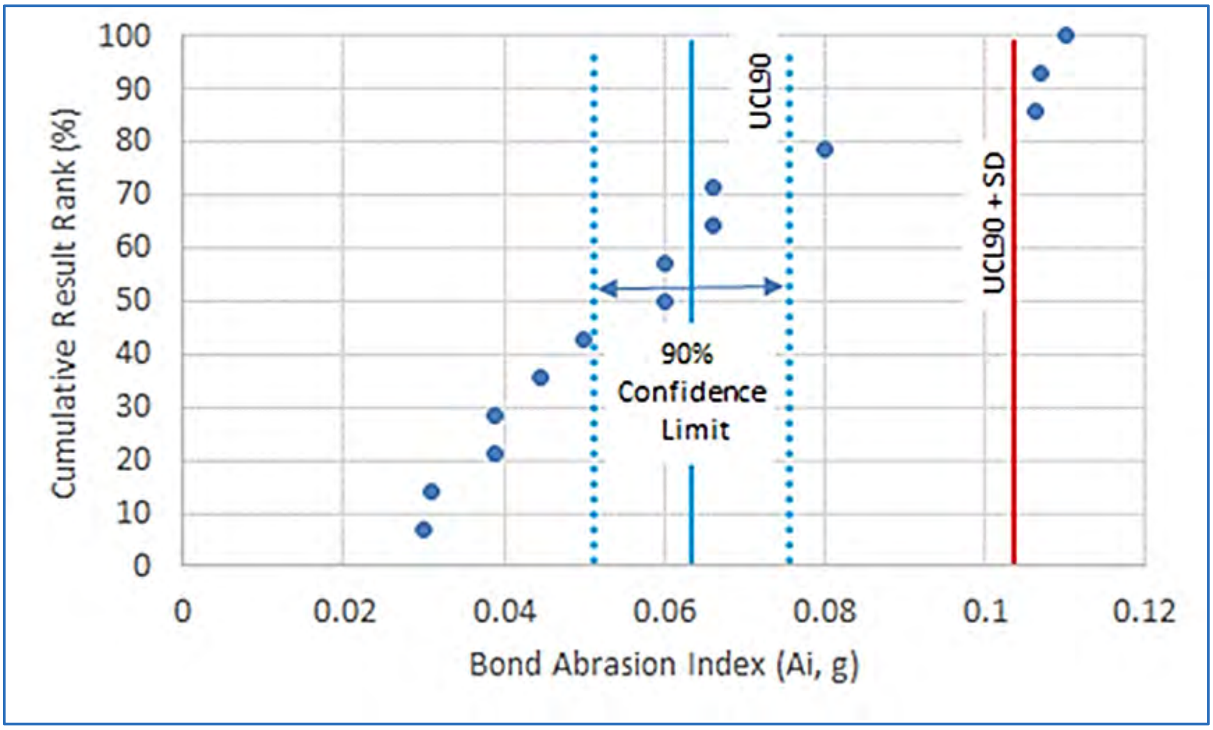
The Kamoa 2017 PFS basis of design (BOD) used the comminution properties in Table 13.12. An appropriately high CWI value has been selected.

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



Amec Foster Wheeler, 2016.

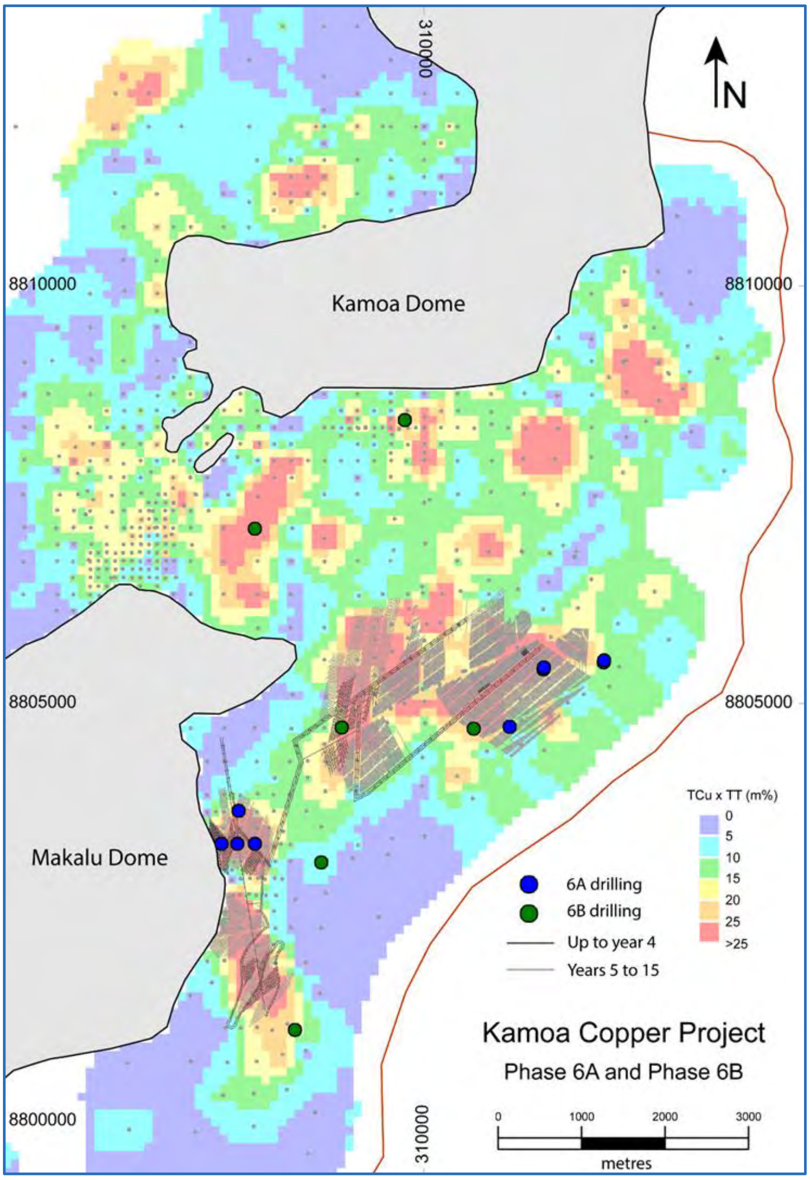
The mean value for the set is Ai = 0.063. The confidence limit is a measure of how confidently the mean or average value has been measured by the testing actually performed. As more samples are tested, the measurement of the mean value improves. Practically speaking, it means that If the same number of samples were chosen and tested again for Ai from all the available samples, then nine 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



Ivanhoe, 2016.

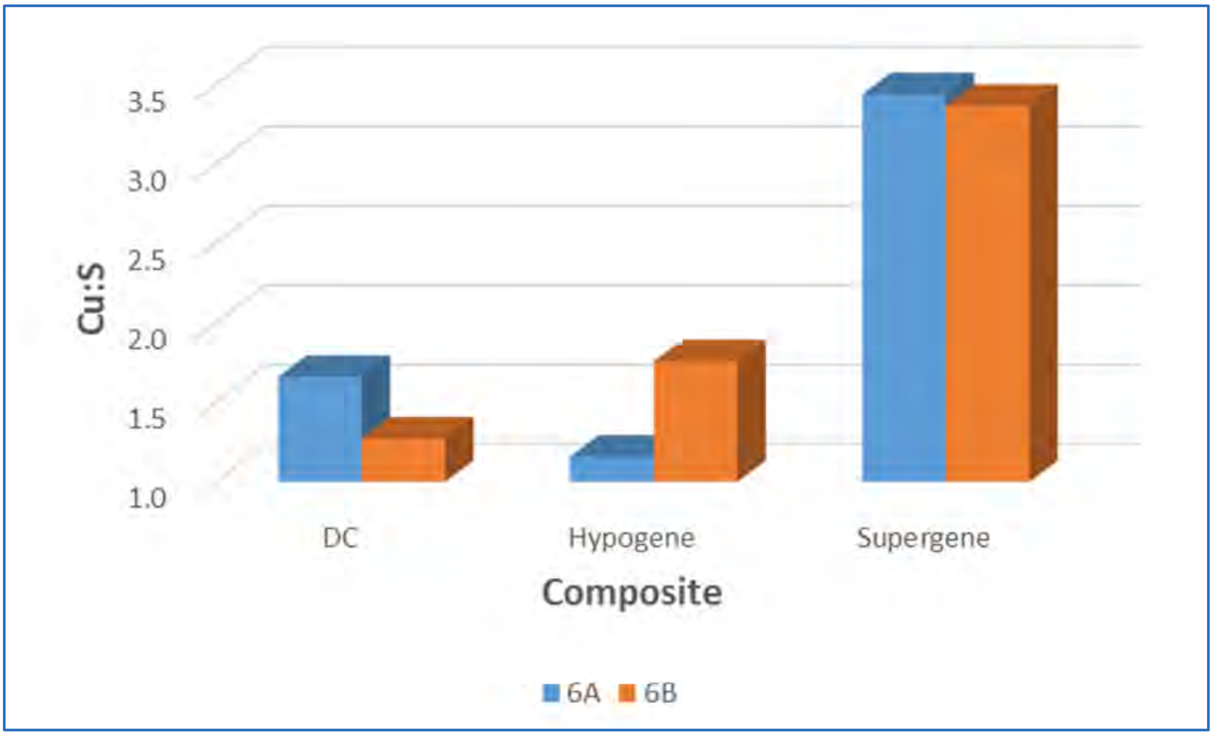
The Phase 6 samples were prepared in sets containing a development composite (DC) and two individual composites based on copper sulfide mineralisation classification. The composite head assays are contained in Table 13.13.

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



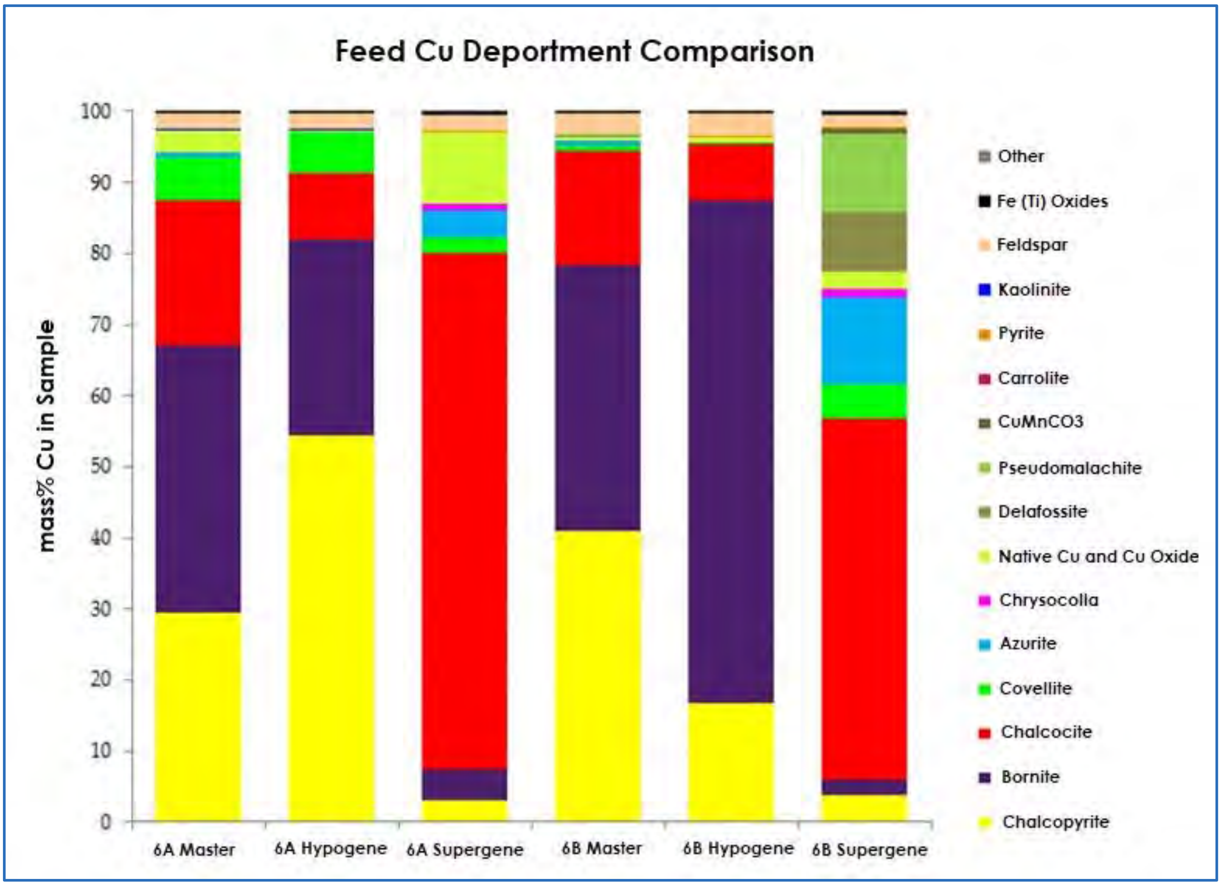
Amec Foster Wheeler, 2016.

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

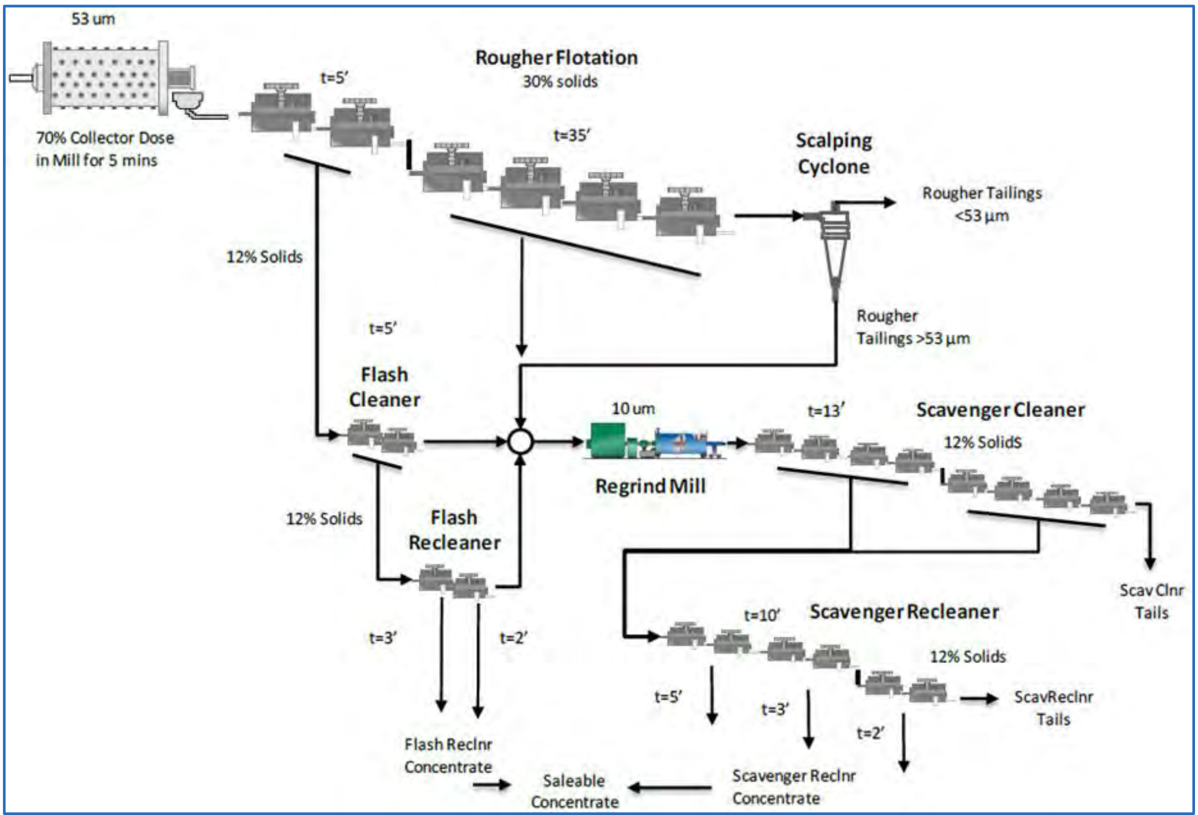


XPS, 2015.

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



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

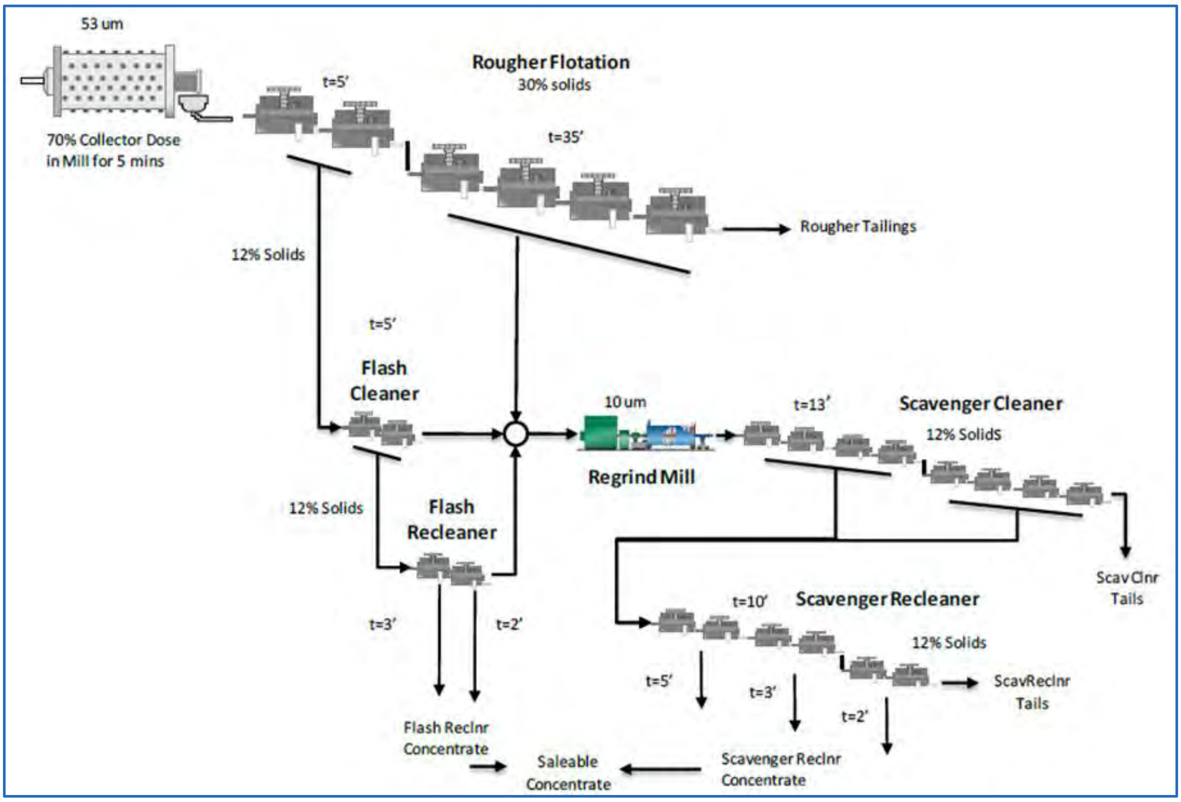
The lack of lime in the test has improved both grade and recovery for the 6A supergene sample. It is notable that the tailings grades are identical and, in general, these two results using the one sample show that the repeatability of the test is excellent.

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



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

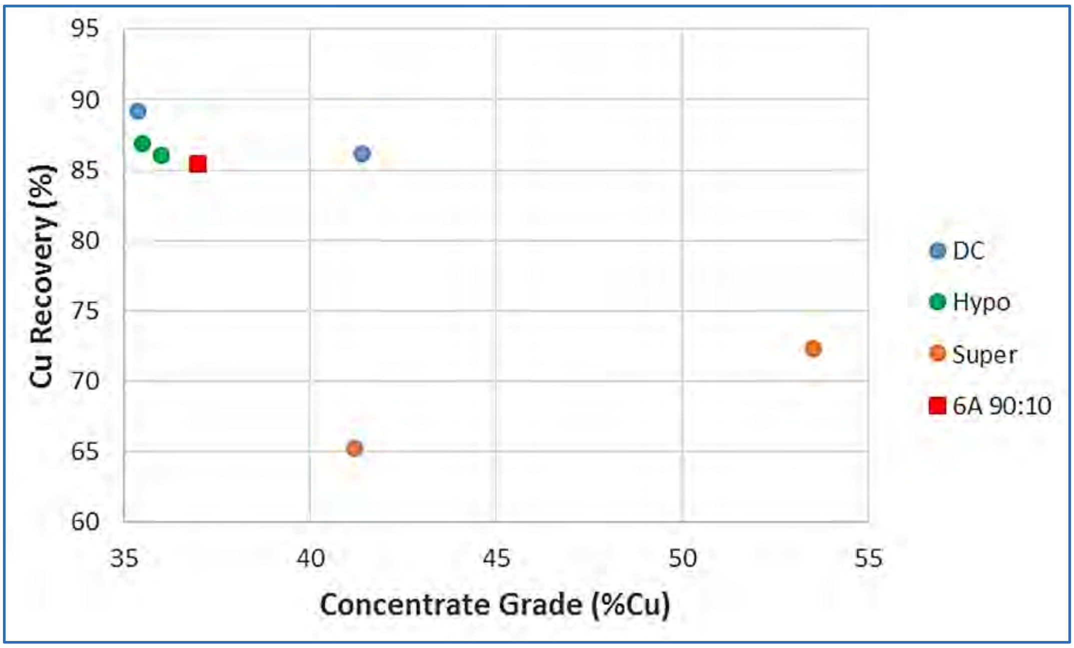
Composite		Final Concentrate					Tail	Feed
		Mass (%)	Cu (%)	Rec Cu (%)	SiO_2 (%)	Fe (%)	Cu (%)	Cu (%)
6A	DC	7.80	41.4	86.2	11.10	16.80	0.56	3.74
	90:10 H: S	8.33	37.0	85.4	6.34	22.00	0.58	3.61
	Hypogene	8.48	36.0	86.1	4.00	21.00	0.54	3.54
	Supergene	5.25	53.5	72.3	13.50	13.40	1.14	3.89
6B	DC	8.07	35.4	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



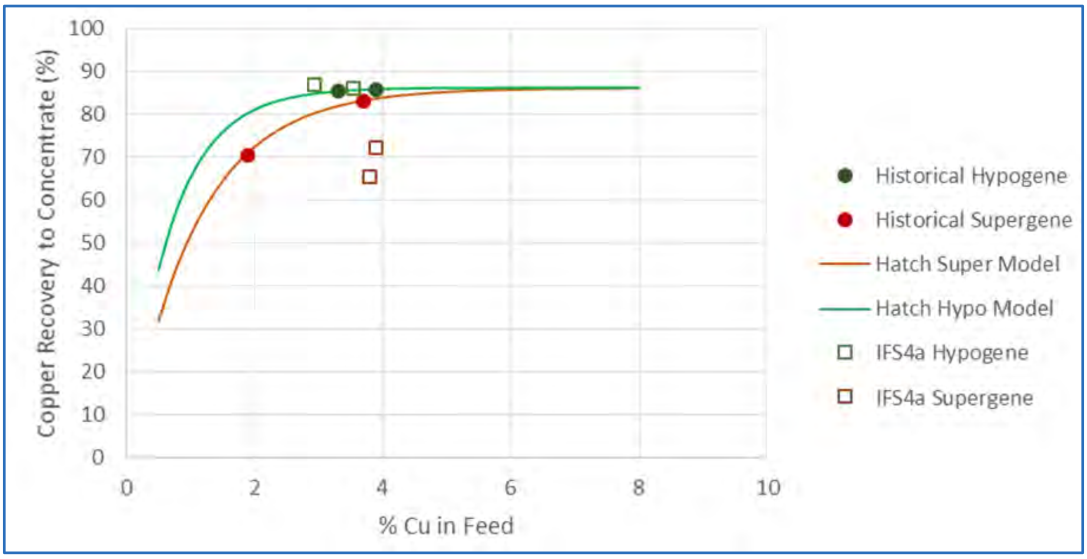
Amec Foster Wheeler, 2016.

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.

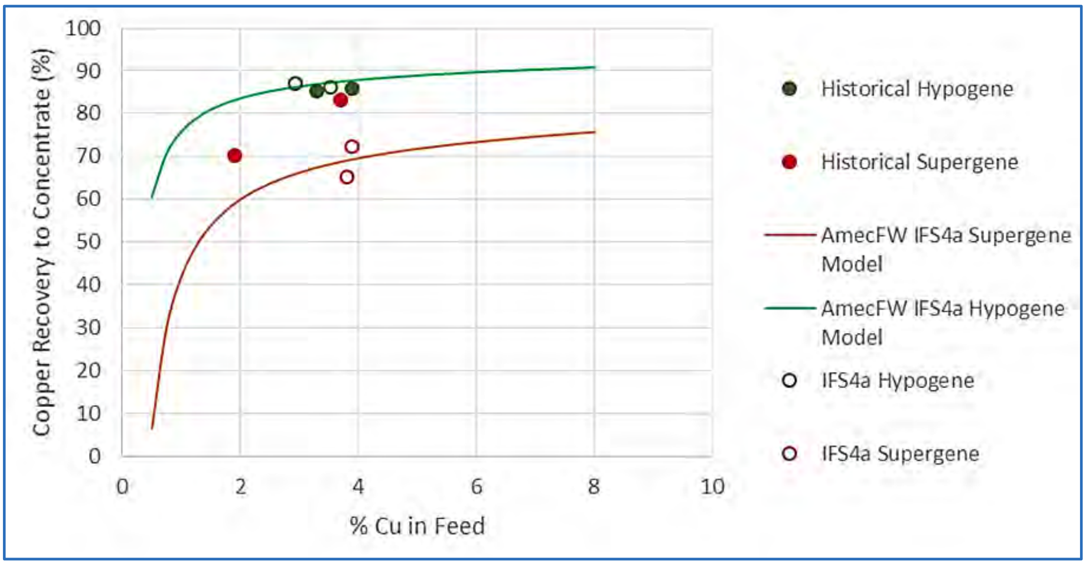
Figure 13.13 Old Copper Recovery Model (TR 2013)



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 2017 PFS Testing



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

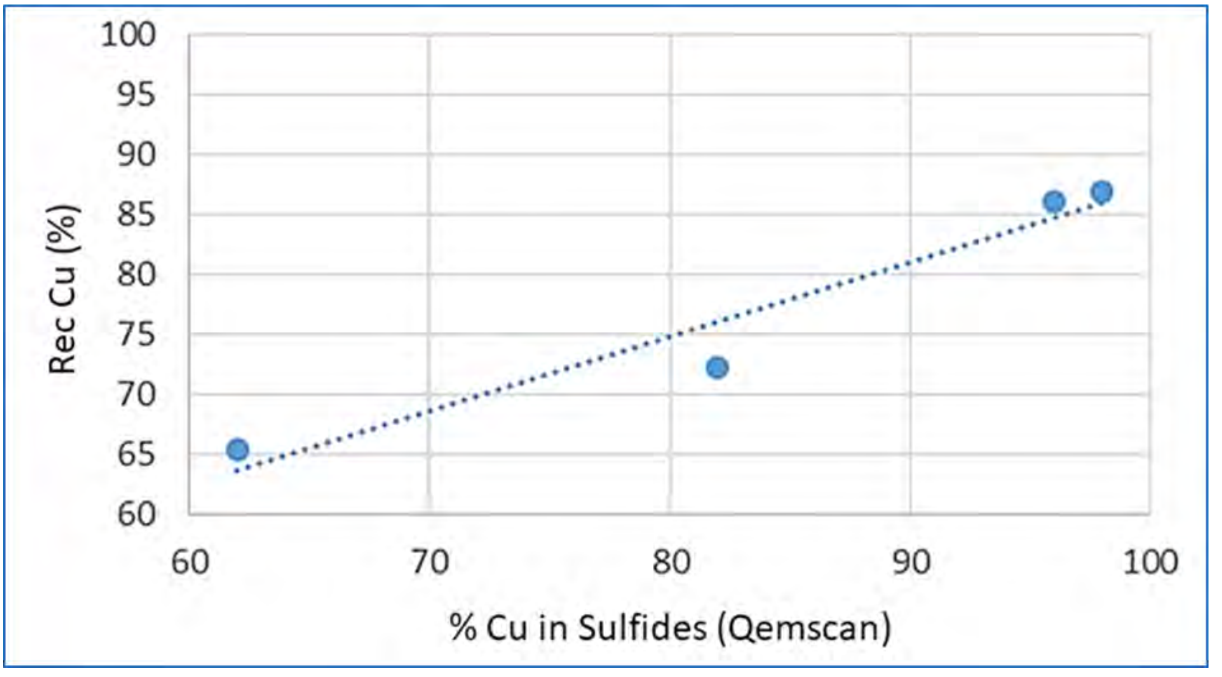
It is clear from Figure 13.14 above that supergene recovery is not well defined when it is necessary to rely on a single dependency, in this case the copper head grade. There will be a recovery relationship with head grade, but the analysis shows that the recovery is more dependent upon the proportion 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).

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



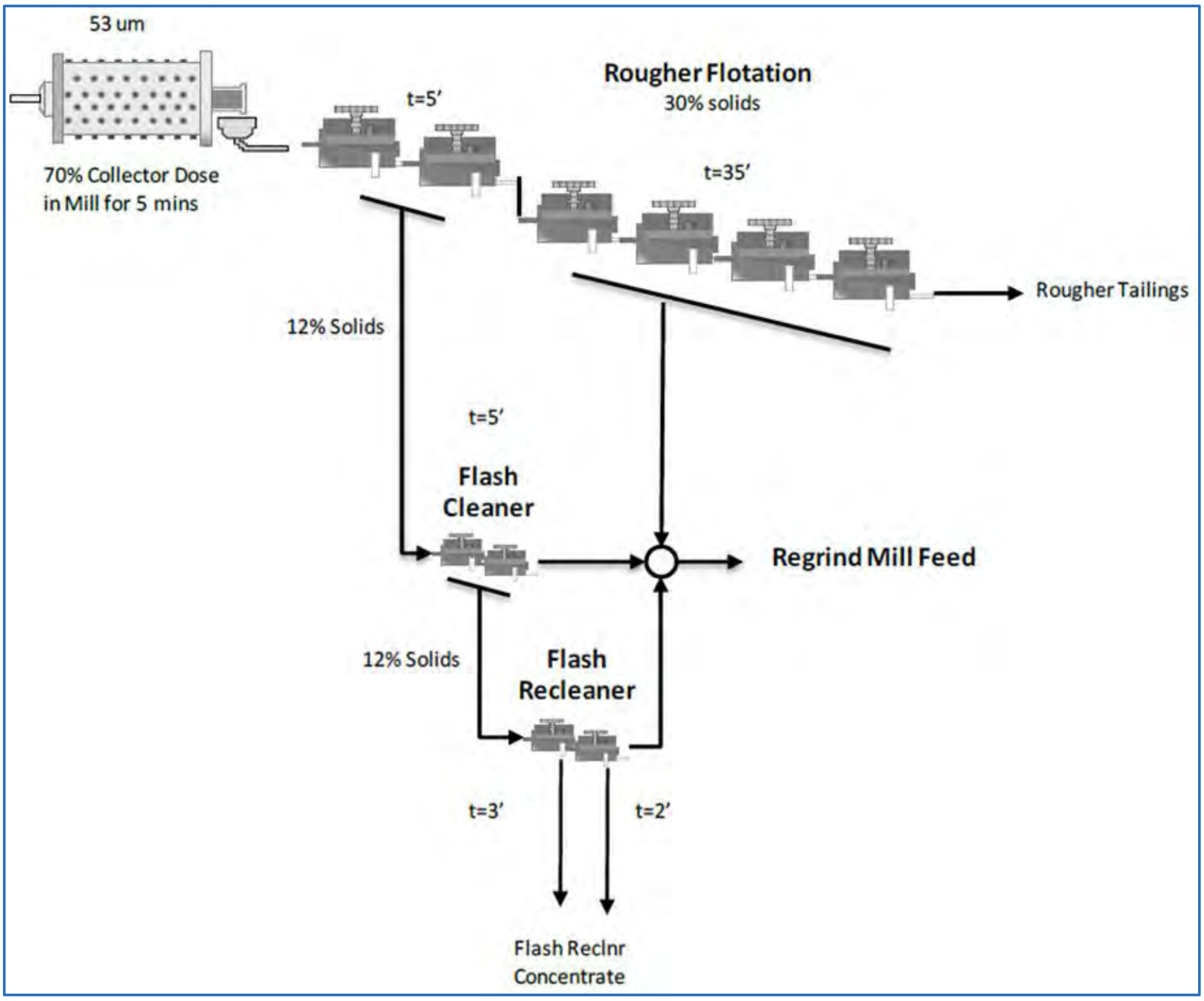
Amec Foster Wheeler, 2016.

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



XPS, 2015.

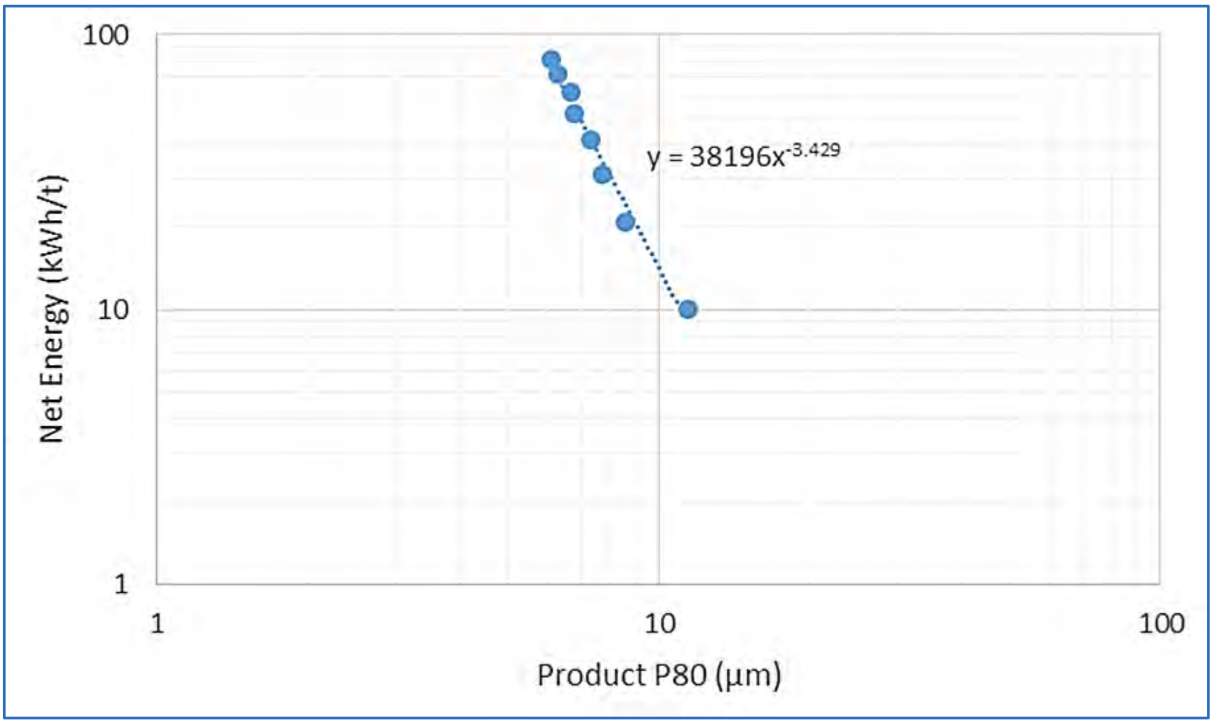
The 6A signature plot composite was prepared separately from the other composites and contained 4.35% Cu. The Cu:S is 1.37 compared to 1.66 for the 6A DC sample indicating a greater proportion of chalcopyrite in the copper mineral suite of the new composite.

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.

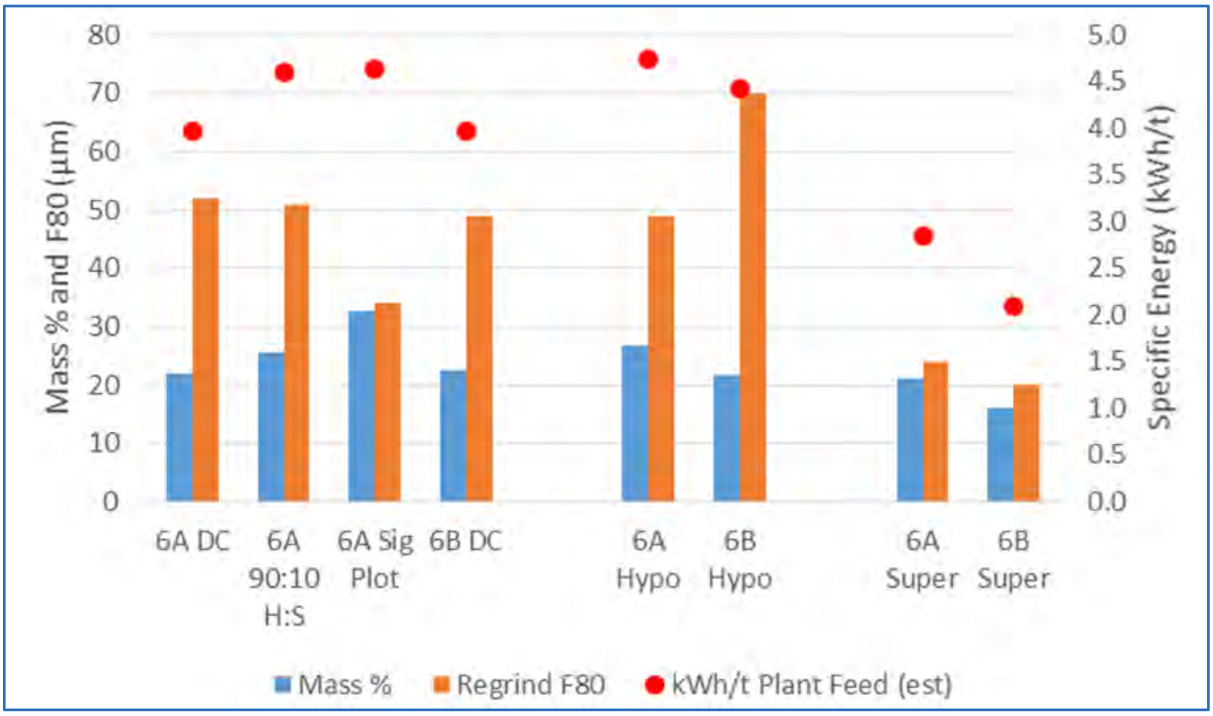
Figure 13.17 IsaMill Signature Plot



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



Amec Foster Wheeler, 2016.

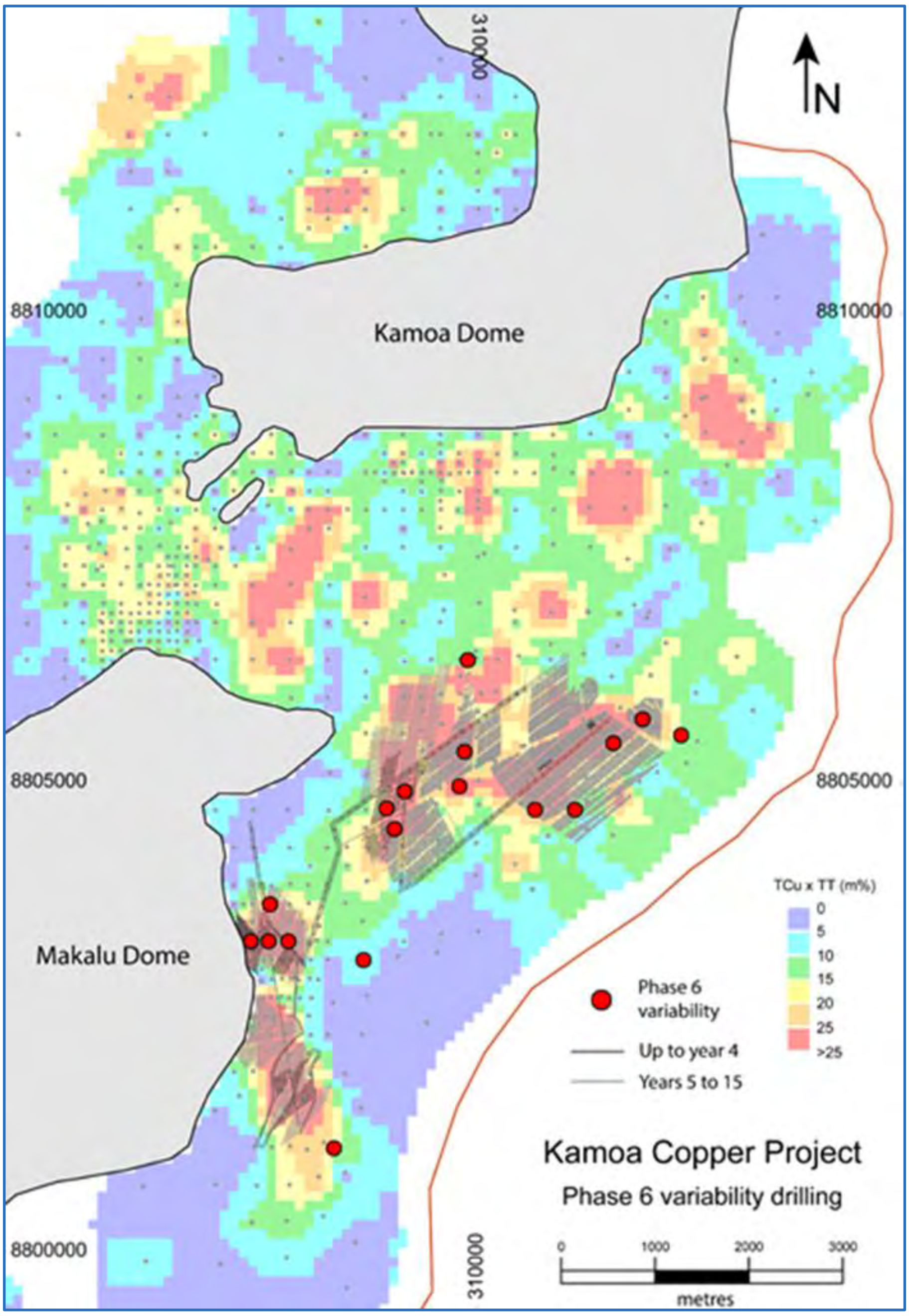
Interestingly, across the four development composites and two hypogene samples the energy per tonne of plant feed is somewhat independent of the test. This is because low mass pulls tend to have coarse particle sizes while high mass pulls are finer. From the Figure 13.18 data, a regrind power selection 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



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.

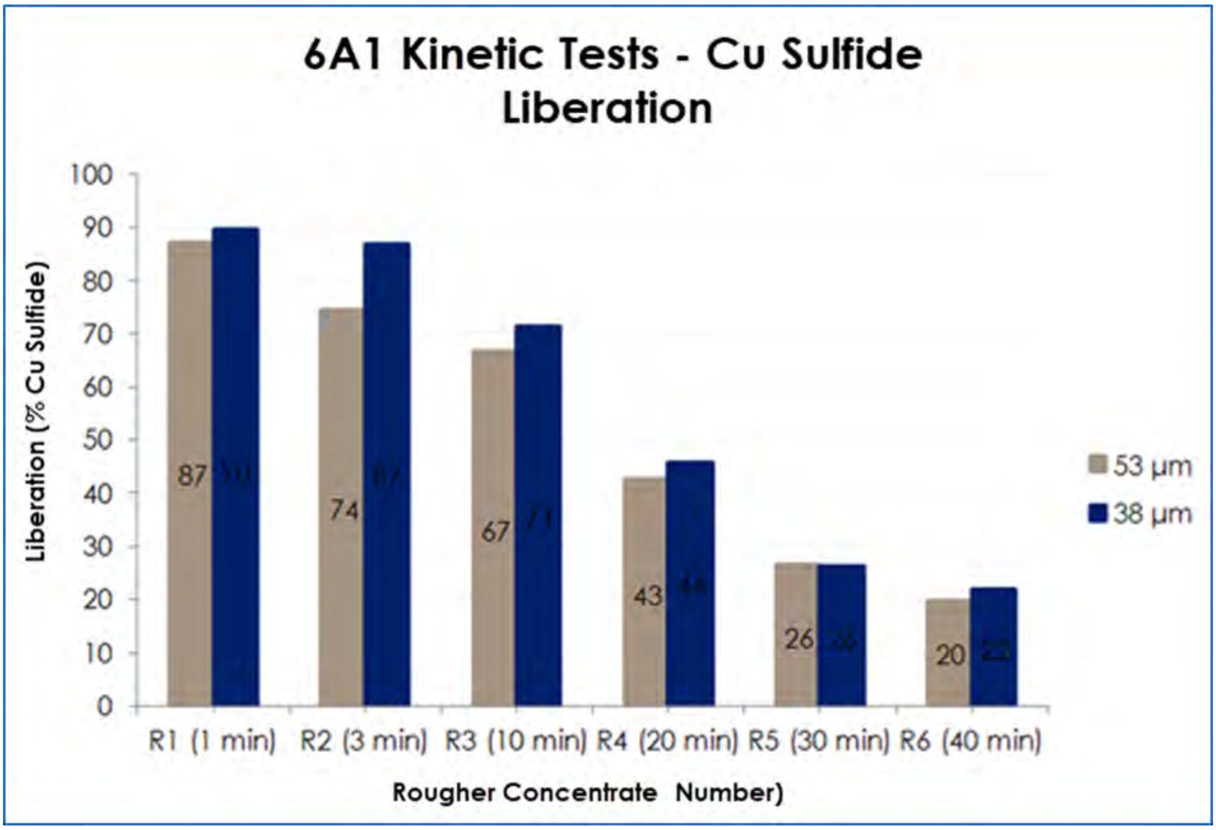
In the image below (Figure 13.20) can be seen a 2 cm wide white clast within the grey diamictite matrix, against which chalcopyrite has "mantled" during the sulfide deposition phase. In the surrounding rock matrix there are smaller mantled clasts and visible blebs of chalcopyrite (and other sulfides). 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

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

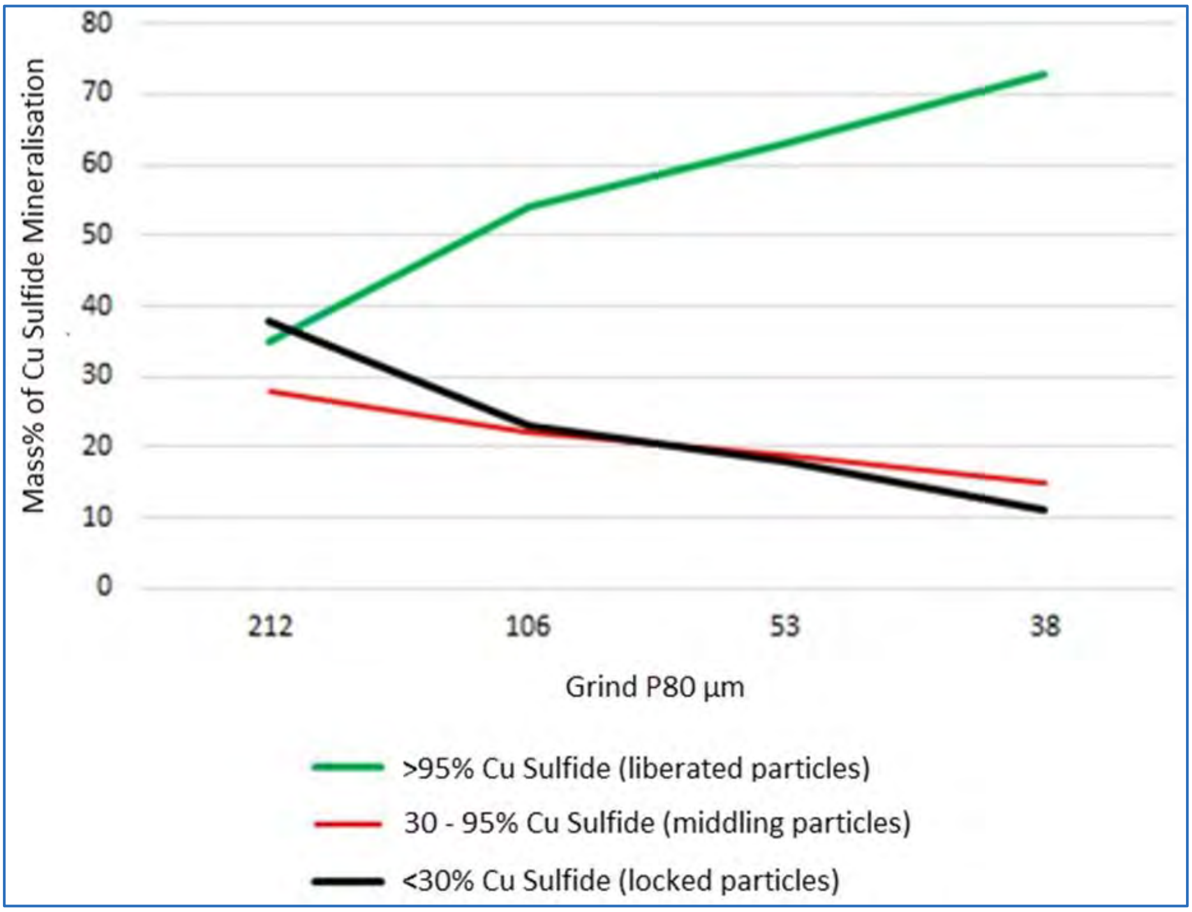


XPS, 2015.

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

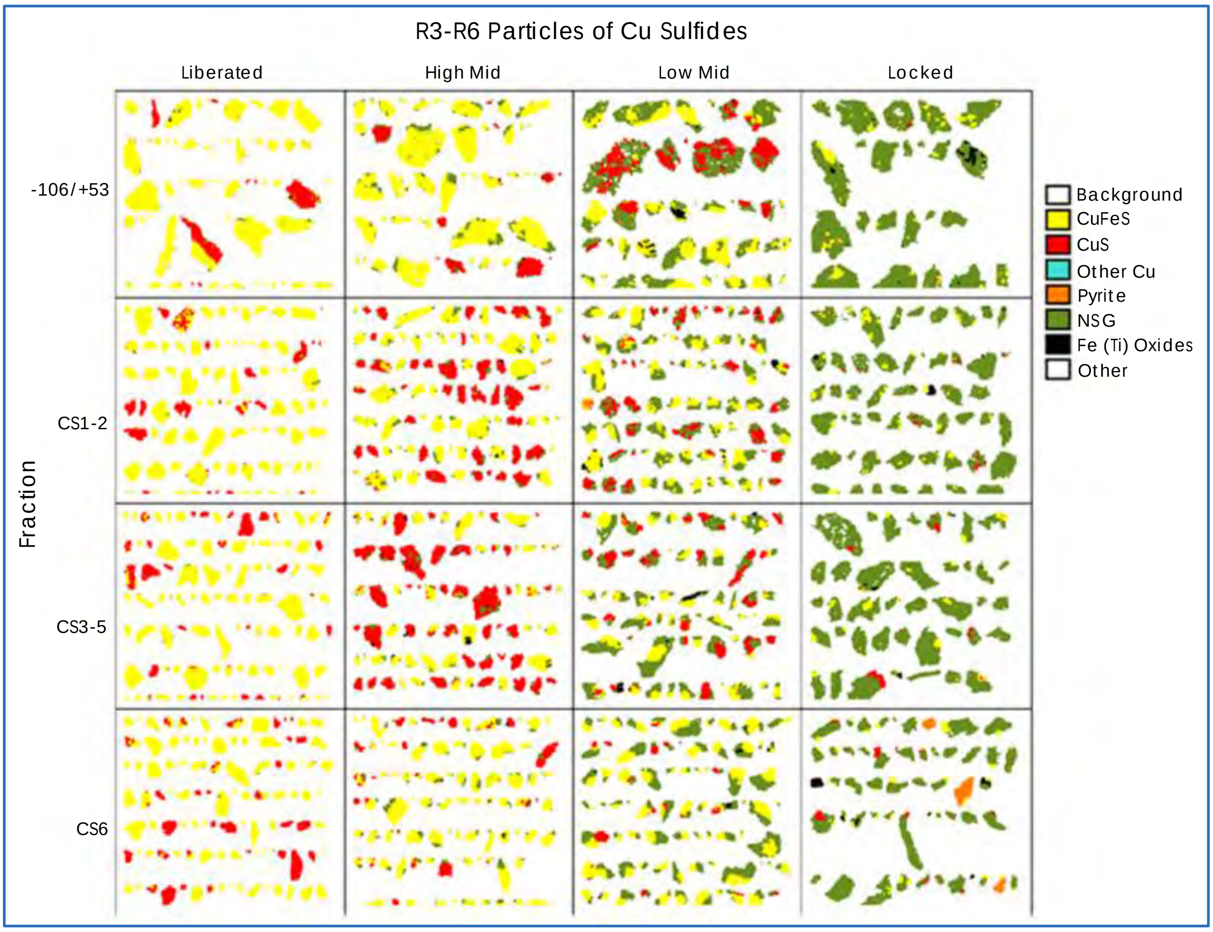


Amec Foster Wheeler, 2018: from XPS data, 2014.

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



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.

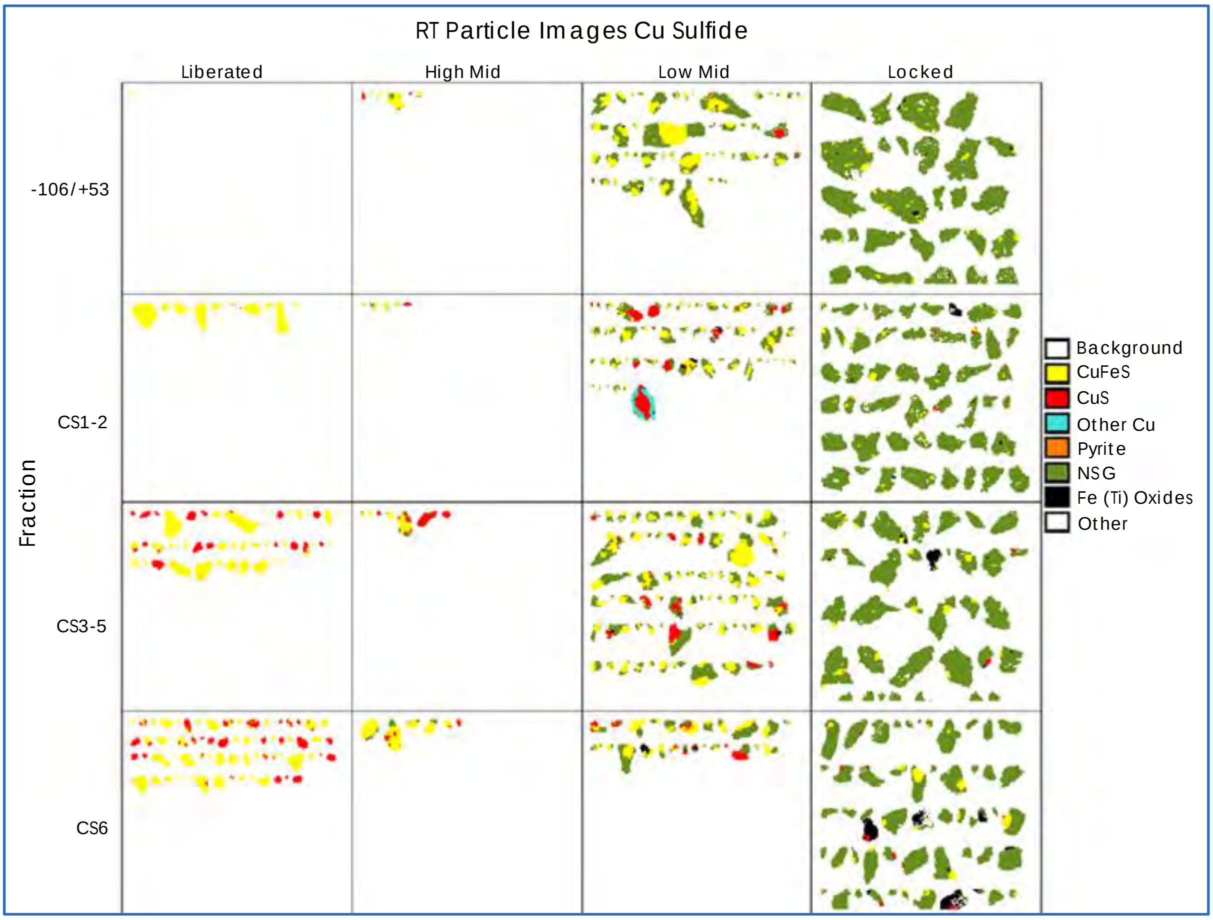
The pervasive fine copper sulfides cause large amounts of attached silicates to be recovered in rougher flotation, and this leads to the high rougher mass pull values (20–40%) typical in the test programmes. At coarse grinds, such as 150 µm P_{80}, large silicate particles invariably have exposed fine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



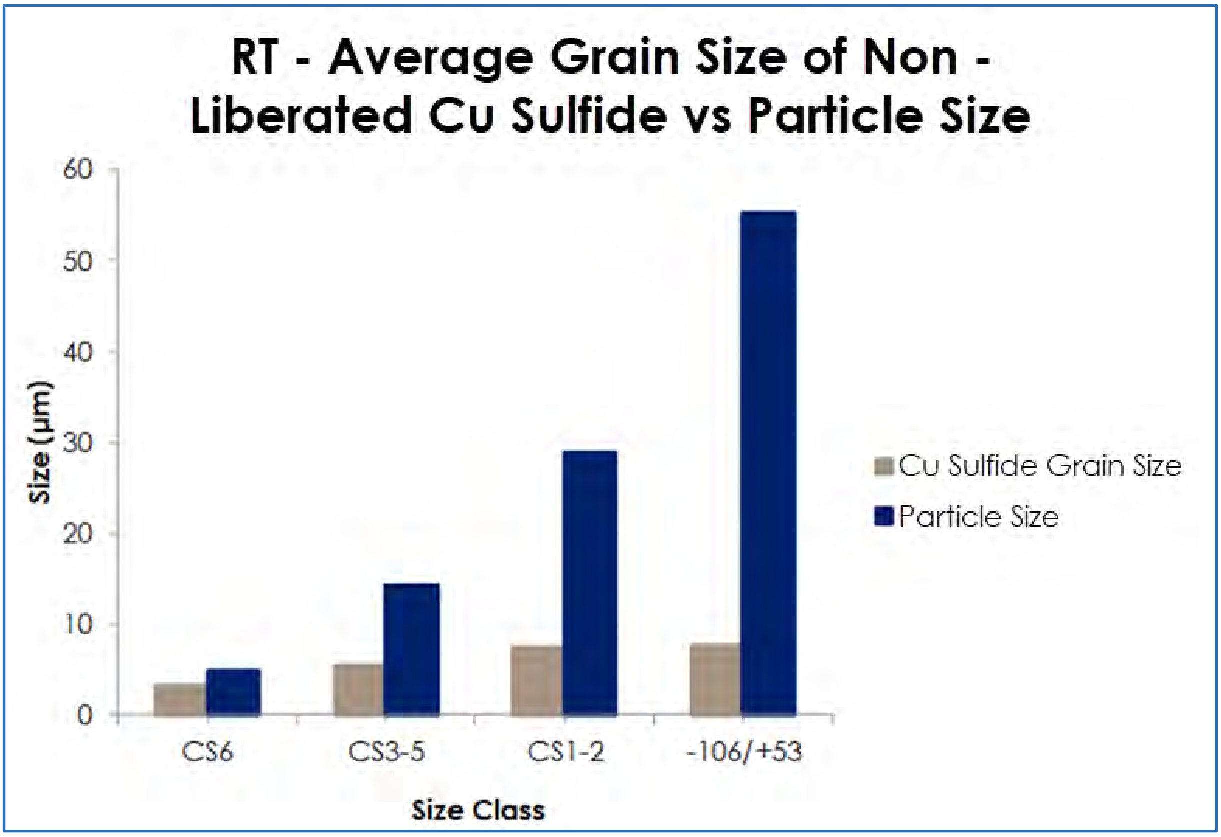
XPS, 2015.

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



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.

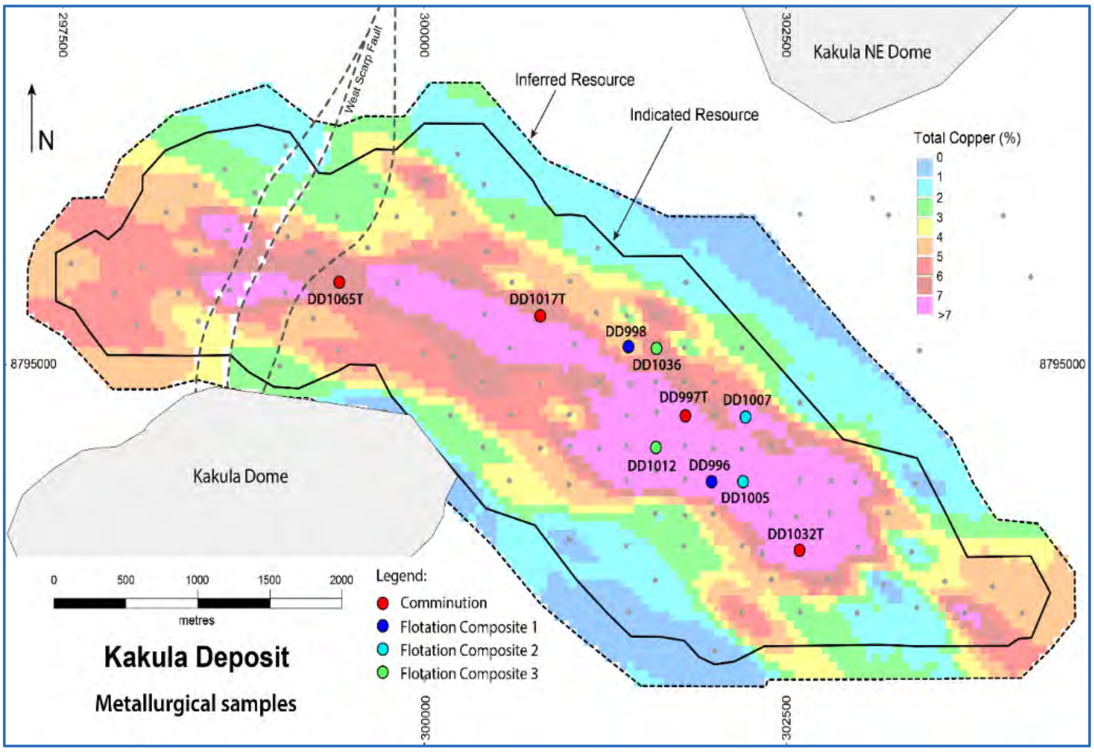
Following successful testing of these early holes, and due to high-grade intercepts consistently achieved at Kakula, additional samples from drillholes DD1005 and DD1007 (Flotation Composite 2) were sent to Zijin laboratories, and DD1012 and DD1036 (Flotation Composite 3) were shipped to XPS to verify 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 analyses 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 (%)
PFS flotation master composite sample	6.13	1.66	56.47	5.16	13.73	1.25	4.10

For the core samples listed above, 10 kg of each were kept aside, during sample preparation, and used in the flotation variability testwork.

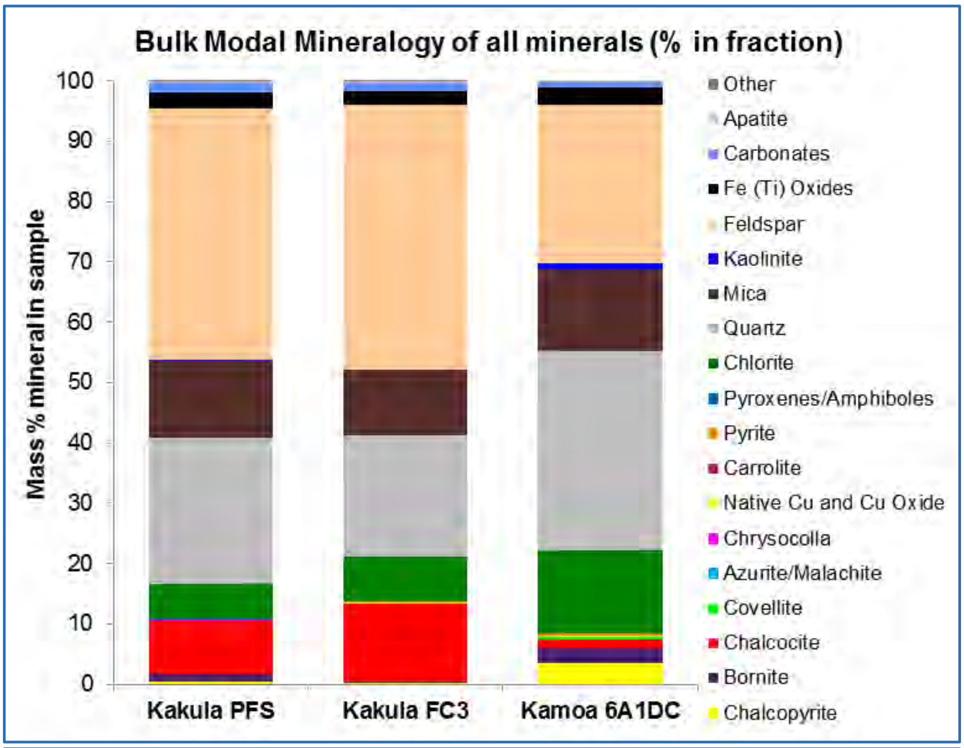
13.3.2 Mineralogical Studies

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



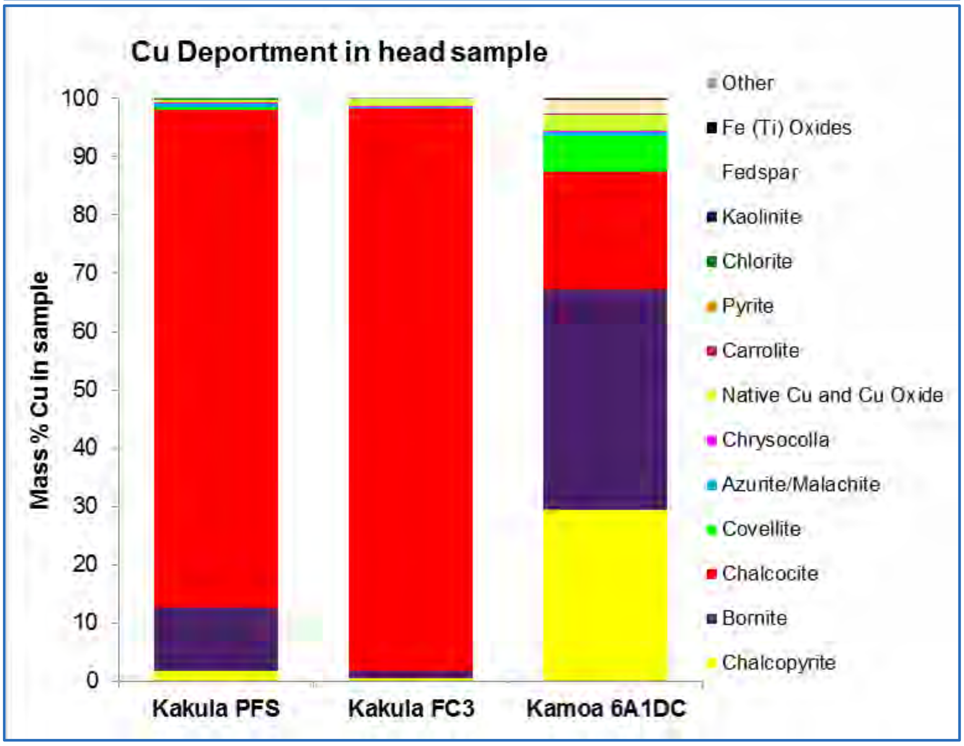
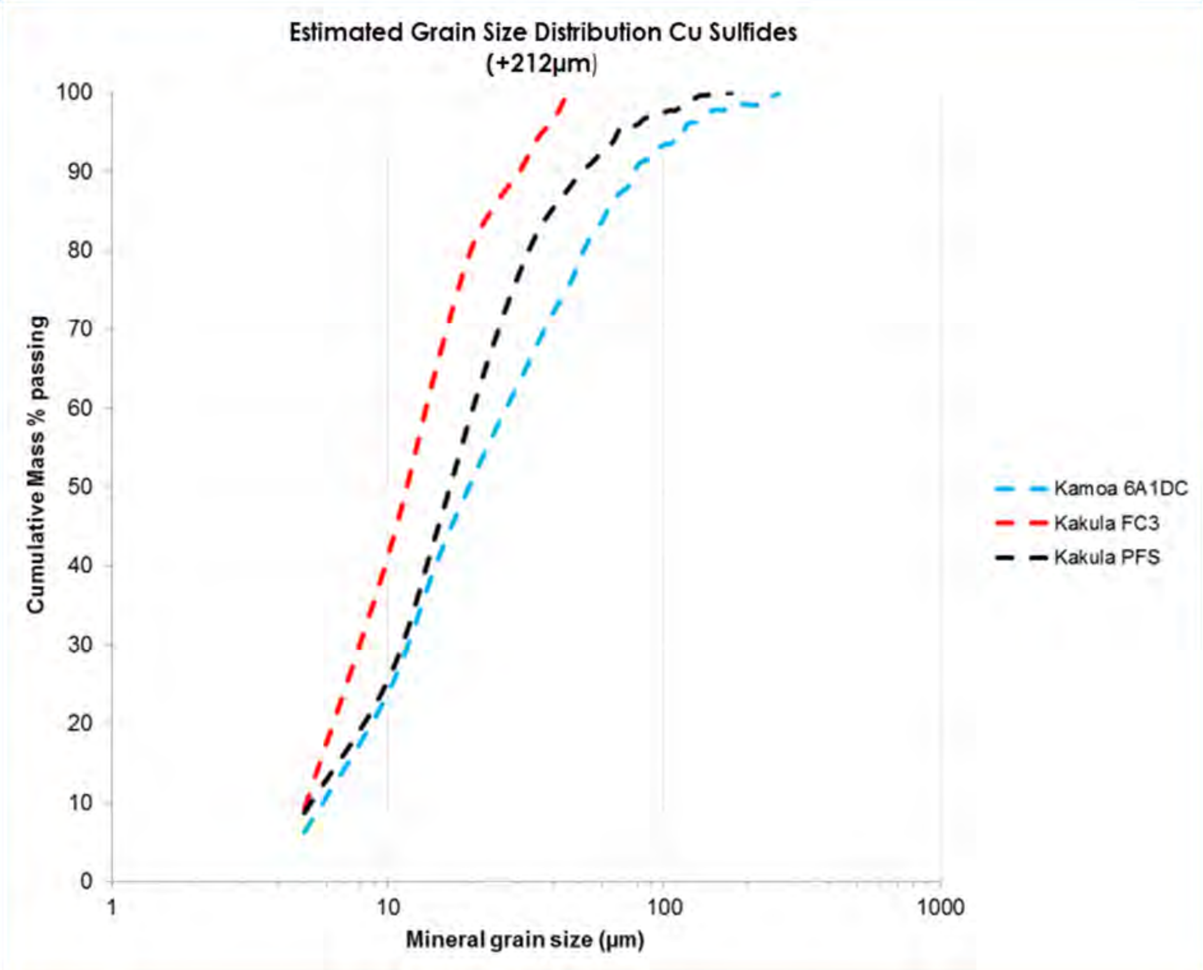


Figure 13.28 Cu Sulfide Grain Size Distribution Comparison Between Kamoa and Kakula



Ivanhoe, 2018.

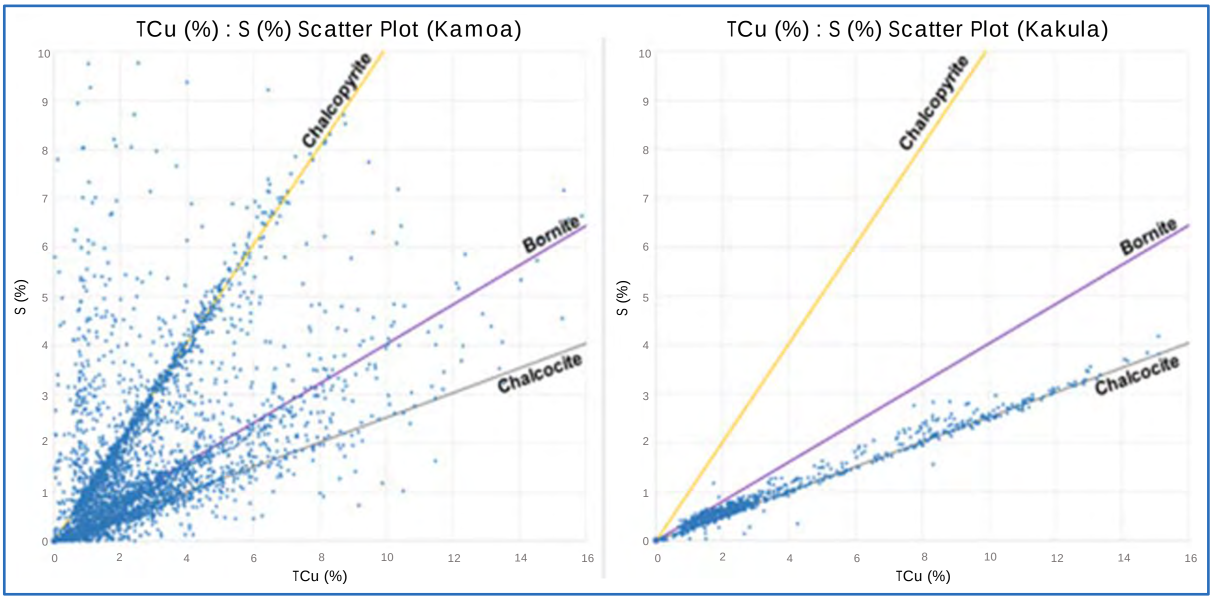
13.3.2.1 Dominance of Chalcocite

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



Ivanhoe, 20116.

13.3.3 Kakula Comminution Testwork

Mintek was contracted by Kamoa Copper SA during 2017 to perform characterisation testwork on diamictite, and siltstone samples from four drill cores (DD1065T, DD1017T, DD997T, and DD1032T). Composite samples, of the diamictite footwall and siltstone footwall, were also tested. Comminution parameter variability testing was also completed in 2018, using samples from drillholes DD1047W1, DD1084W1, DD1021W2, DD1061W1, DD1070W1, and DD1145W2.

The scope of work included:

- Uni axial compressive strength (UCS)
- Bond crushability work index (CWi) and drop weight tests (DWi)
- SAG mill comminution (SMC)
- Bond abrasion index (Ai)
- Bond rod work index (BRWi)
- 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/m³ for the diamictite samples, and 10.1 kWh/m³ 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 1700 rpm.
- Air addition method was changed from forced too 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

Sample	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.

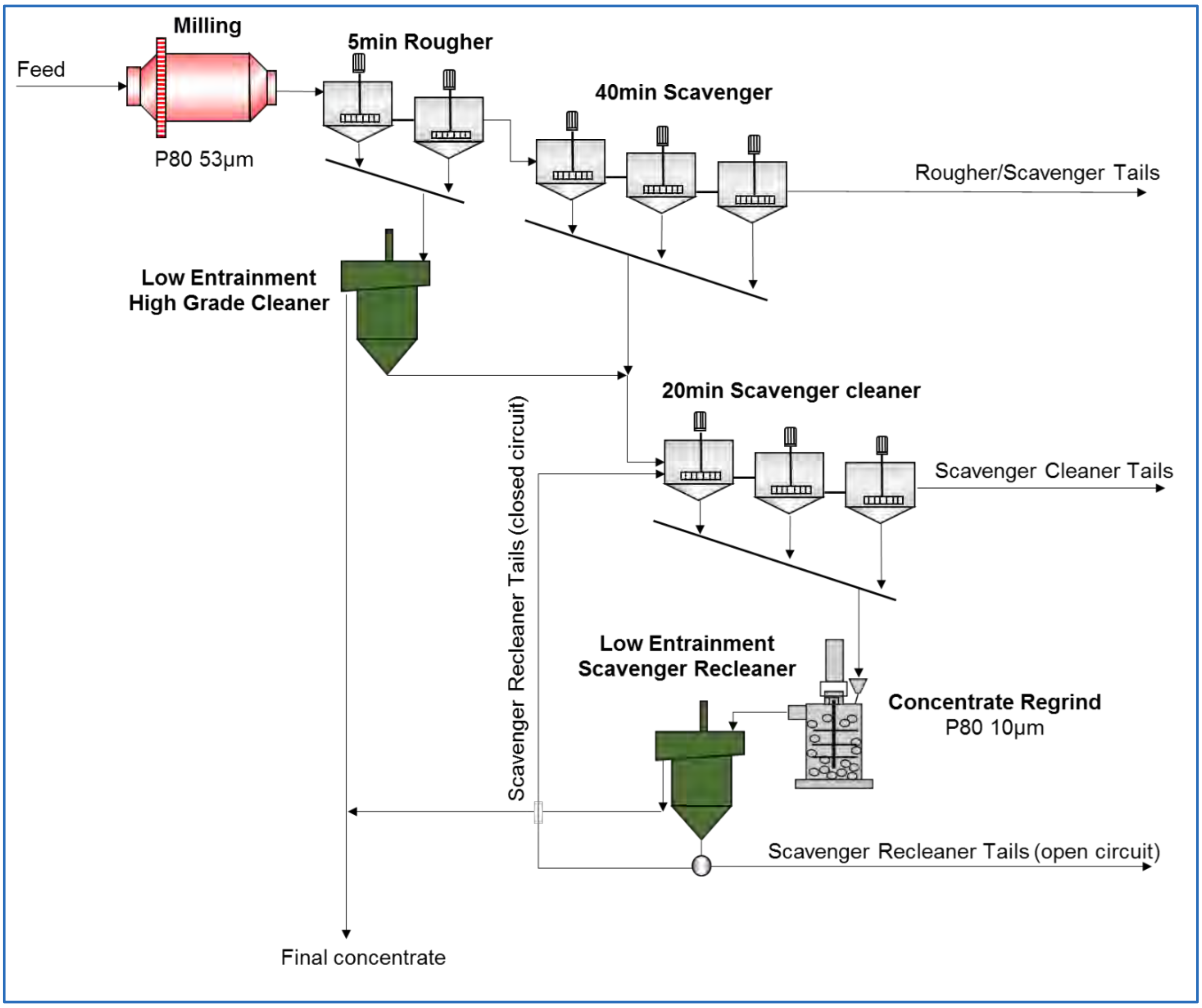
- Re-positioning of the regrind step, from the scavenger cleaner feed to the scavenger recleaner feed, reduces the mass reporting to the regrind circuit from 30% to 12% of the fresh feed. A small increase in collector addition, to the scavenger recleaner stage as well as an increase in scavenger recleaner residence time from 10 min to 18 min was needed to improve recleaner recovery kinetics. Shifting of Cu units from the high-grade circuit to the scavenger cleaner circuit did not improve scavenger cleaner unit recoveries.
- Low entrainment cleaning tests was conducted to determine if the concentrate grades could be increased by reducing the amount of gangue carried over to the concentrate by means of entrainment. Better selectivity of Cu over Silica was achieved in the concentrate.

13.3.5.3 Kakula 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.

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

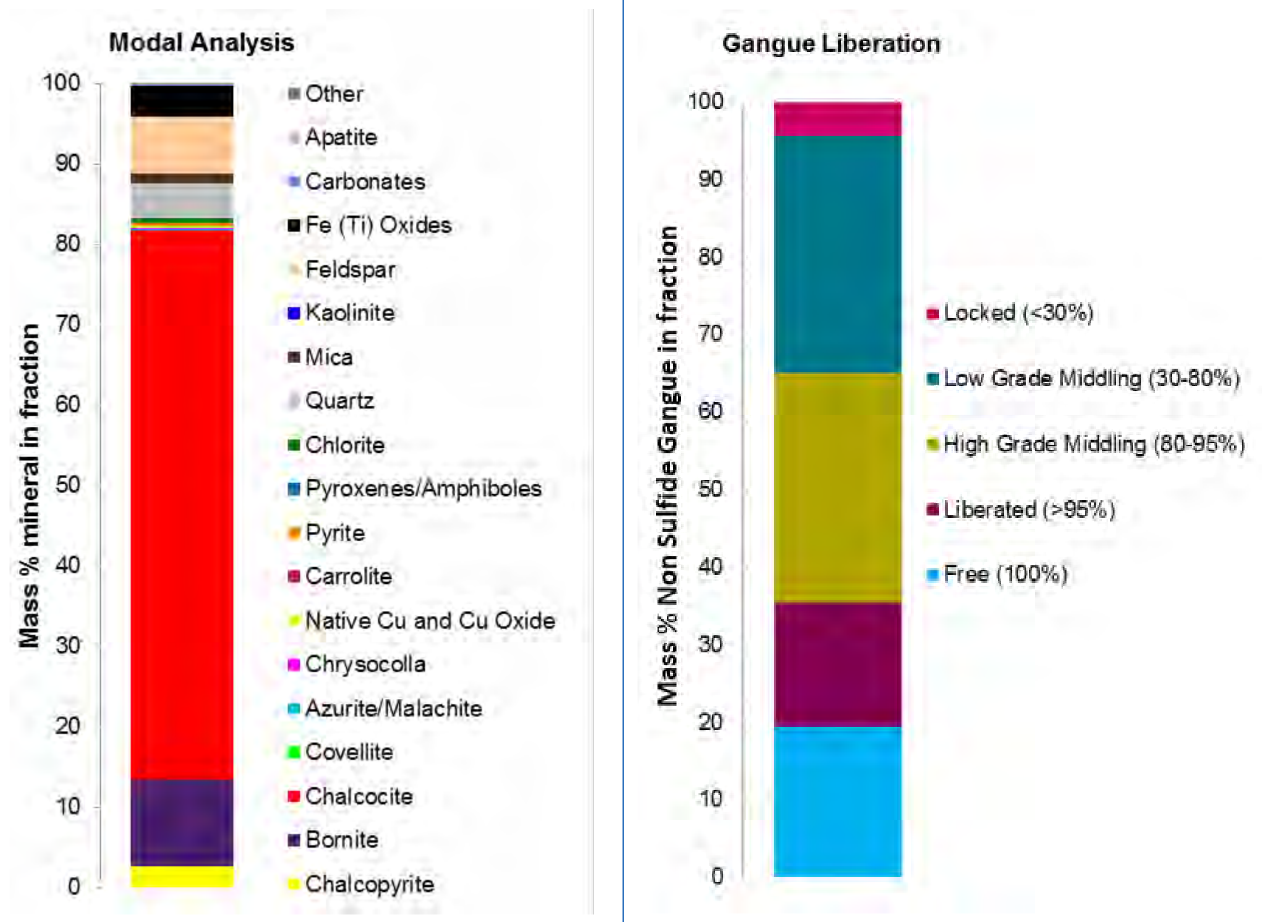


Table 13.25 Kakula PFS Concentrate Analysis

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
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 ultra-fine 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

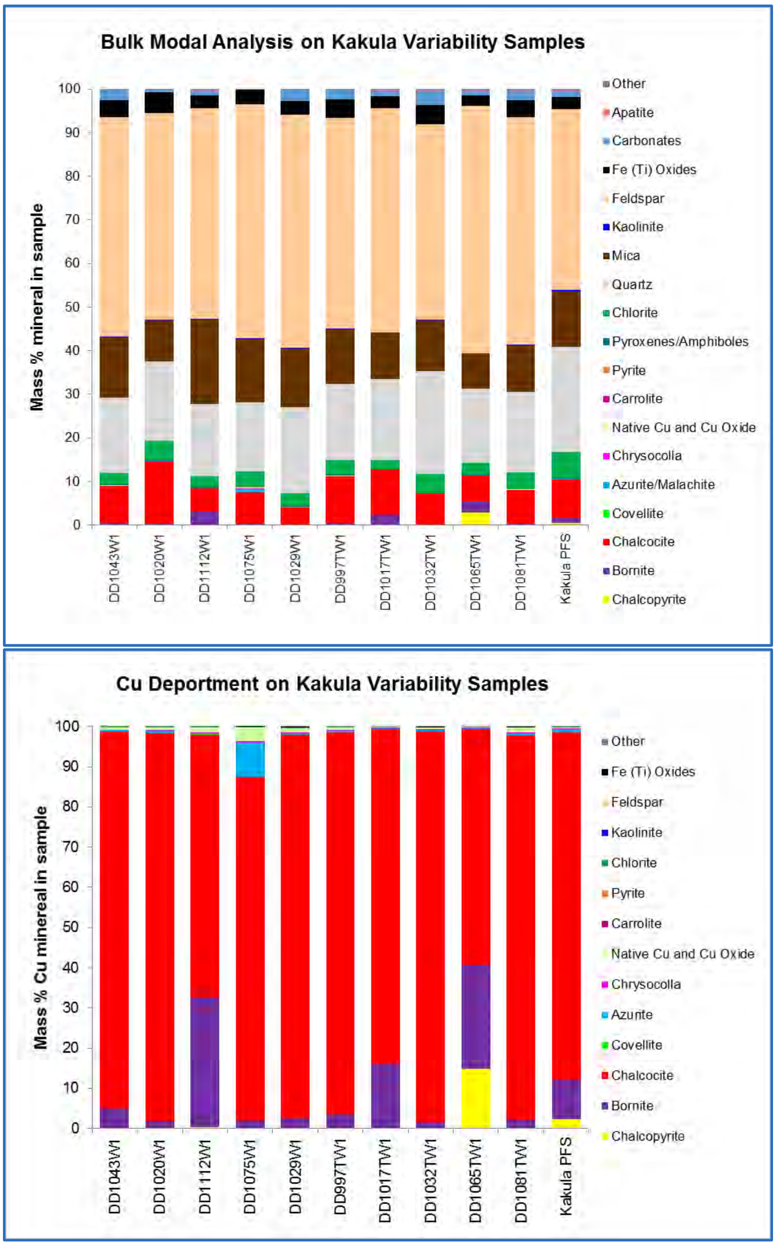
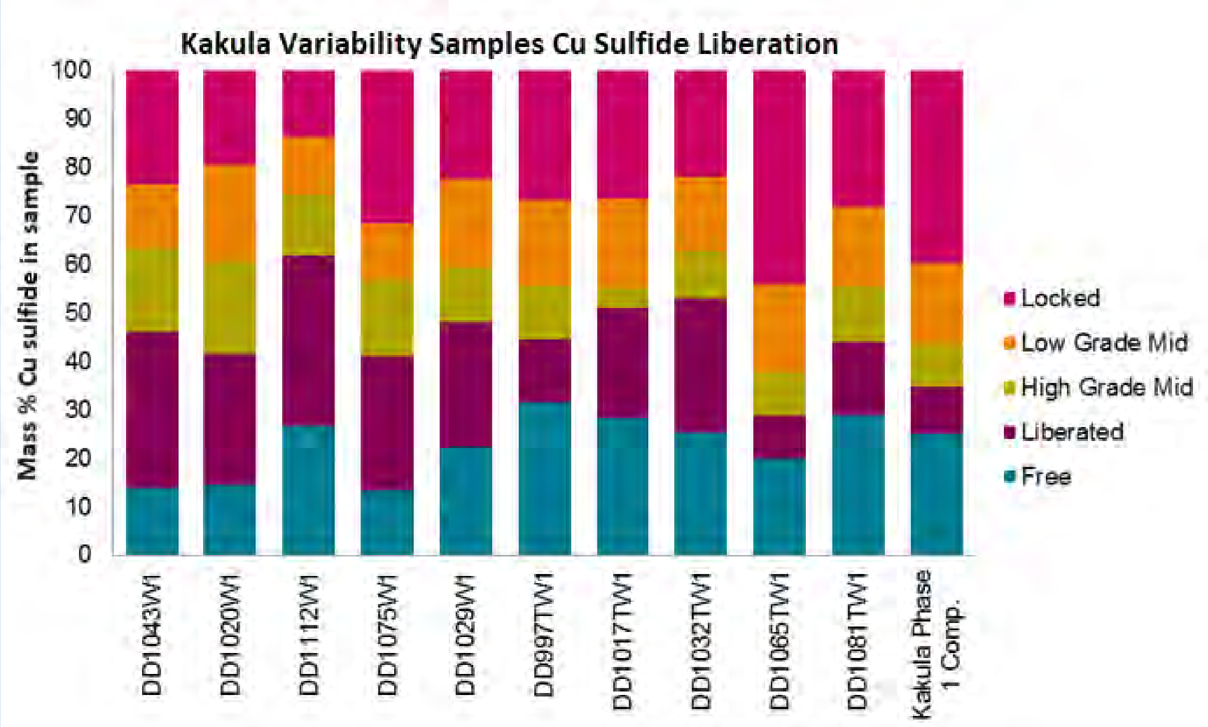


Figure 13.33 Kakula Preliminary Flotation Variability Samples Liberation Data



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 50 g/t total collector for each percentage of Cu in the head – to allow for changes in head grade across the samples. The ratio of SIBX:Areo3477 was maintained at 85%:15%. Refer to Table 13.27 for a summary of the final concentrates produced in each of the tests.

Table 13.27 Kakula Preliminary Flotation Variability Results

Sample	Mass Pull (%)	Head Cu (%)	Cu Recovery (%)	Final Concentrate Product		
				Cu (%)	SiO_2 (%)	Fe (%)
DD1043W1	6.5	6.0	79.0	70.0	4.5	1.9
DD1020W1	11.2	9.2	84.3	67.5	5.1	3.2
DD1112W1	7.1	5.2	90.8	65.9	4.2	5.0
DD1075W1 (Average)	4.8	5.3	64.7	70.6	5.4	2.2
DD1029W1	3.0	2.6	86.9	73.3	2.8	1.5
DD997TW1	7.6	6.7	81.9	73.1	4.5	2.1
DD1017TW1	10.7	8.0	87.9	70.3	4.1	2.6
DD1032W1 (2)	5.7	5.0	82.1	68.4	6.4	2.9
DD1065W1 (Average)	9.5	5.6	81.0	47.2	13.8	10.5
DD1081W1	6.8	5.3	86.3	64.5	8.5	3.1

The data indicated that the chalcocite rich samples produced similar results with Cu recoveries over 80% and SiO_2 grades below 10%. The sample rich in chalcopyrite (DD1065W1) only achieved an average grade of 47% Cu product at 81% Cu recovery, and high SiO_2 at 13.8%. Sample DD1075W1 was elevated in non-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 were:

- 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 –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 were 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/m^2.h. The overflow clarity achieved, with a flocculant dosage of 30 g/t, was 216 mg/l solids to overflow, while the overflow clarity improved to 137 mg/l solids to the overflow with a flocculant dosage of 40 g/t. A yield strength of 99 Pa was measured at a solids underflow concentration of 72.5%.

13.3.7.4 Concentrate Filtration Testwork

Following the thickening testwork, Outotec conducted further testwork on the Kakula PFS concentrate sample to determine the suitability of the Larox® Pressure Filter (PF) and Fast Filter Press (FFP) technology for dewatering of the material. Bench scale testing was conducted to evaluate filter cloth selection, filter cake thickness, filtration rate, cake moister content, and filter cake handling characteristics. A summary of the findings is presented in Table 13.30.

Table 13.30 Kakula PFS Final Concentrate Filtration Testing Results Summary

Dewatering technology	Test	Air drying time (Minutes)	Filtration rate (kg DS/m^{2}h)	Cake Moisture (% w/w)	Cake thickness (mm)	Pressing pressure (Bar)	Air pressure (Bar)
PF Pressure Filter	1	3.0	840	5.8	52	16	9
PF Pressure Filter	2	1.0	1037	7.7	52	16	9
FFP Fast Filter Press	3	3.0	510	6.7	58	12	9

This testwork indicated that the Kakula PFS final concentrate product could be successfully dewatered to within the targeted moistures (8–10%), at high solid flux rates.

13.3.7.5 Tailings Thickening, Rheology and Filtration Testwork

During June 2018, SGS Canada conducted solid-liquid separation, rheology, and pressure filtration testwork on a Kakula PFS final tailings composite sample, prepared as part of the flotation flow sheet development campaign by XPS. The aim of the testing was to determine the optimum thickener design and operating parameters. The testing included material characterisation, flocculant selection, static settling, and batch dynamic thickening. The material characteristics as determined by SGS are presented in Table 13.31 and compared to the Kamoa Phase 6 tailings sample.

Table 13.31 Kakula PFS Flotation Tailings Characteristics (SGS)

Parameter	Kakula PFS Tailings	Kamoa Phase 6 Tailings
Slurry pH	7.8	7.2
Slurry P_{80}	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.

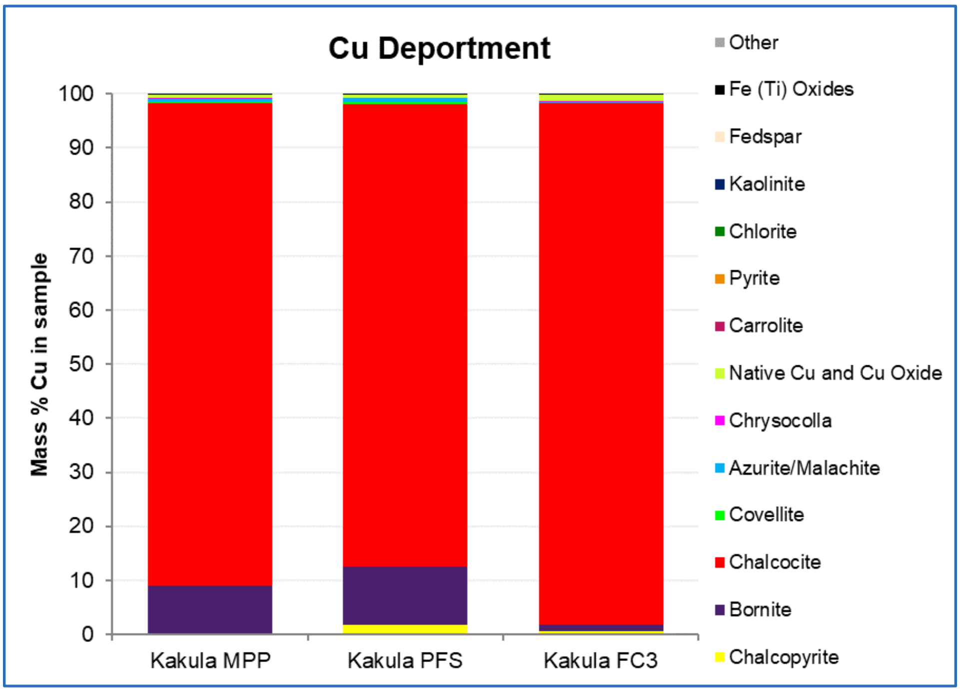
The above mini-pilot plant testwork further demonstrated the performance of the flow sheet developed during the pre-feasibility study. Open circuit batch tests, locked cycle testing, and mineralogical analyses were completed to match the metallurgical performance of the mini pilot plant sample to the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

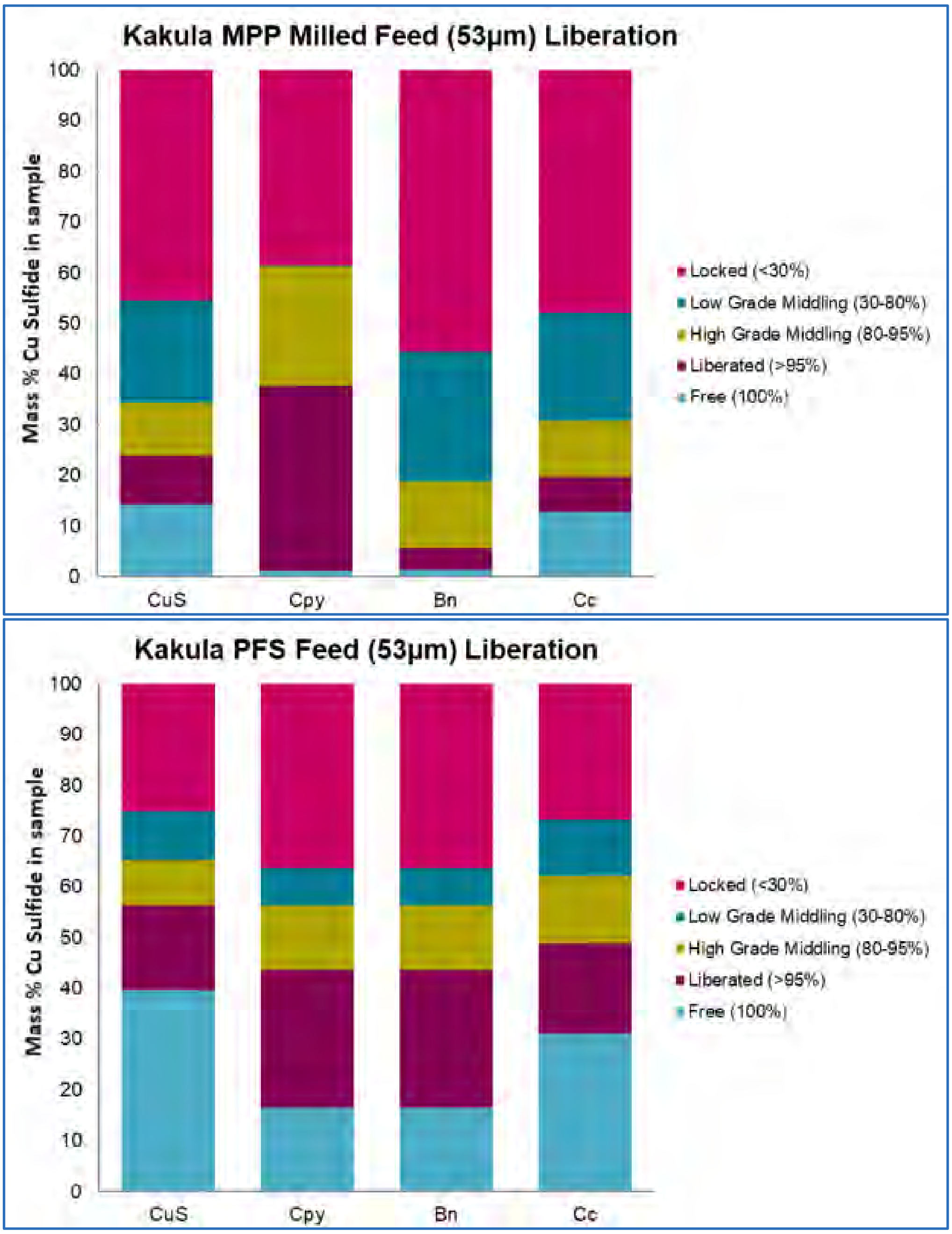


The following was noted:

- 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%.

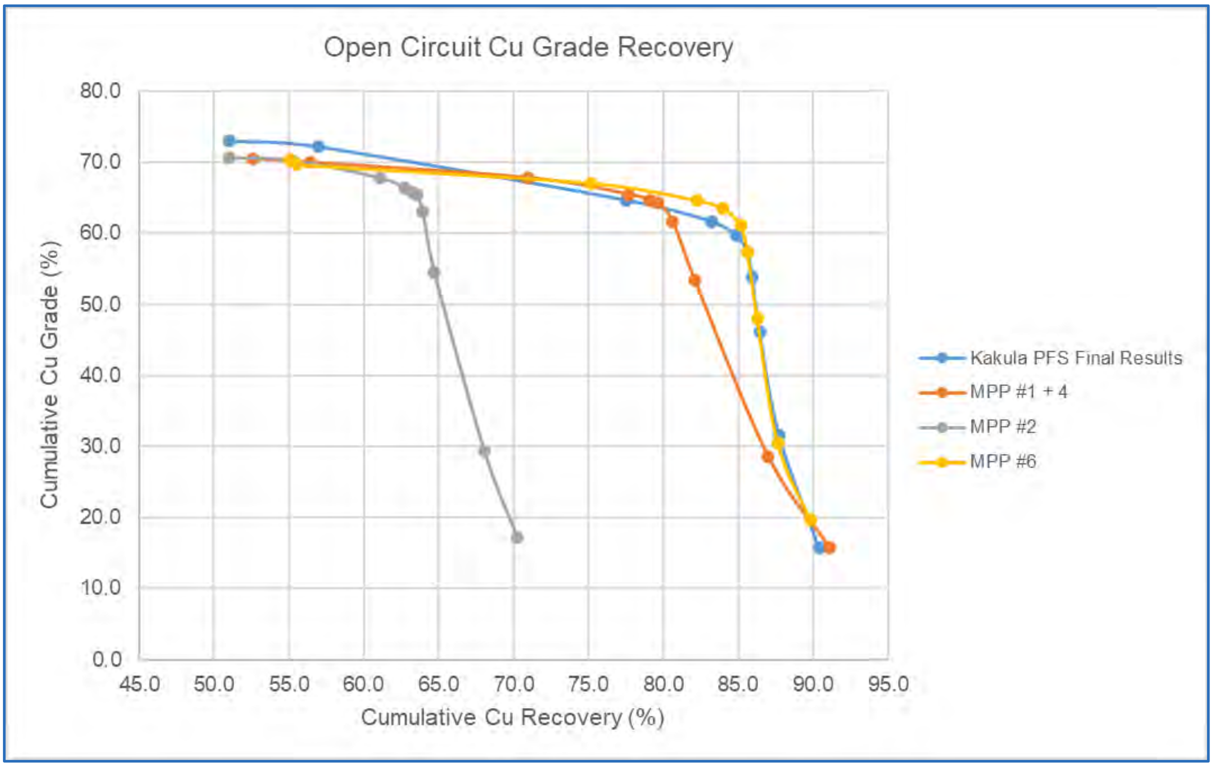
Open Circuit Cleaner Testwork

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 13.26). 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).

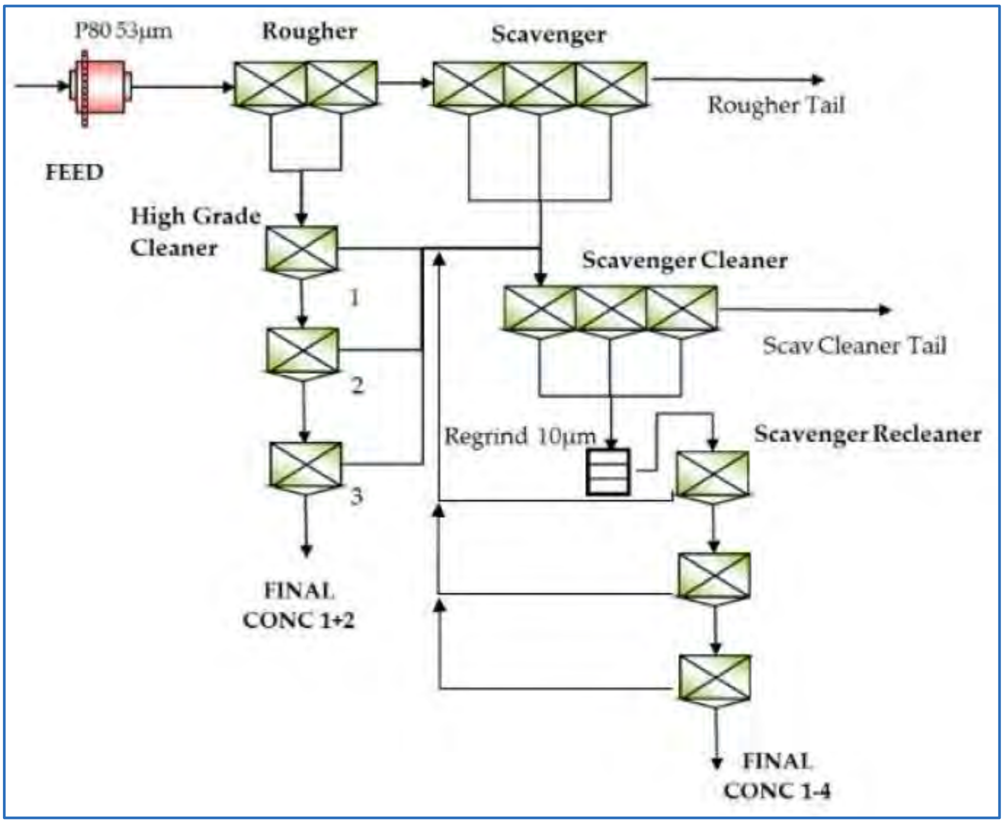
Figure 13.36 MPP Cleaner Circuit Testing Compared to PFS Results



Locked Cycle Testing

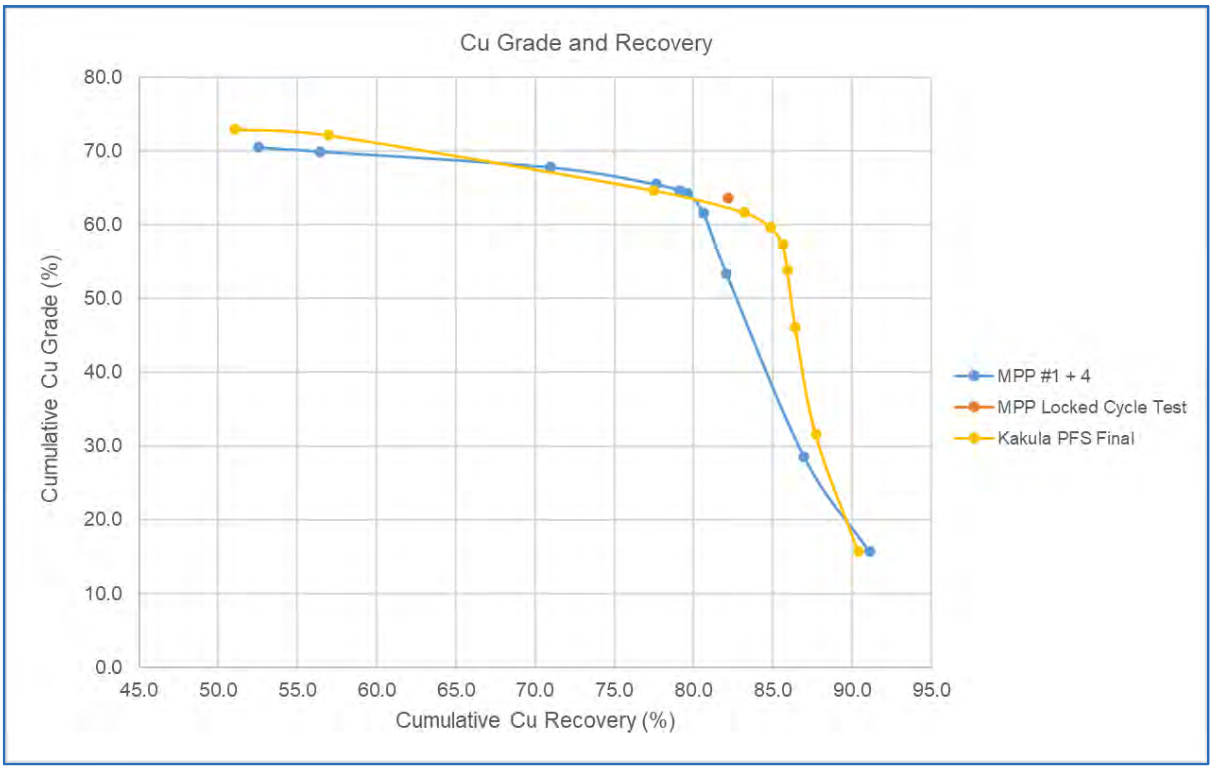
A single, six-cycle locked cycle test was conducted to determine the effect of recirculating the scavenger recleaner tailings back to the scavenger cleaner (Figure 13.37).

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.

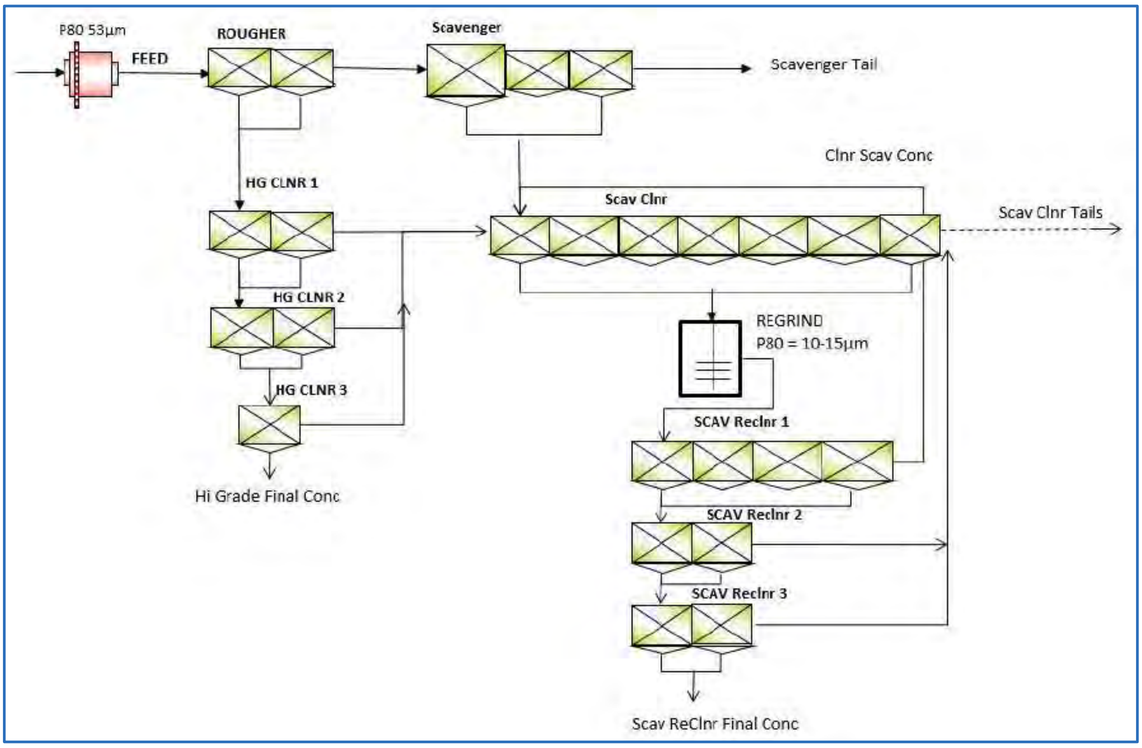
Figure 13.38 MPP Locked Cycle Result Compared to Open Circuit Testing



Backfill Tailings Sample Generation

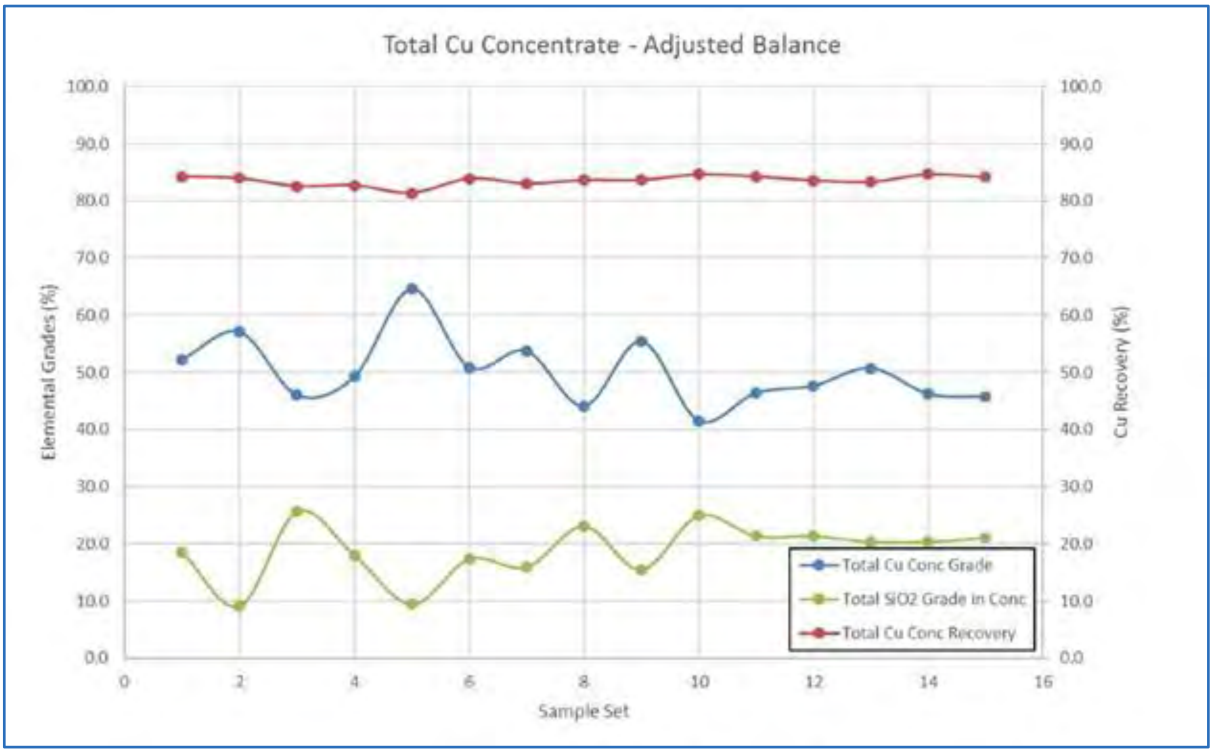
The first MPP run was aimed at producing a combined tailings product for backfill testwork utilising the flow sheet in Figure 13.39.

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



MPP run #1 ran for roughly 58-hours, feeding just under 600 kg of fresh feed. Final products were sampled every four hours 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



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-hours 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 four hours 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

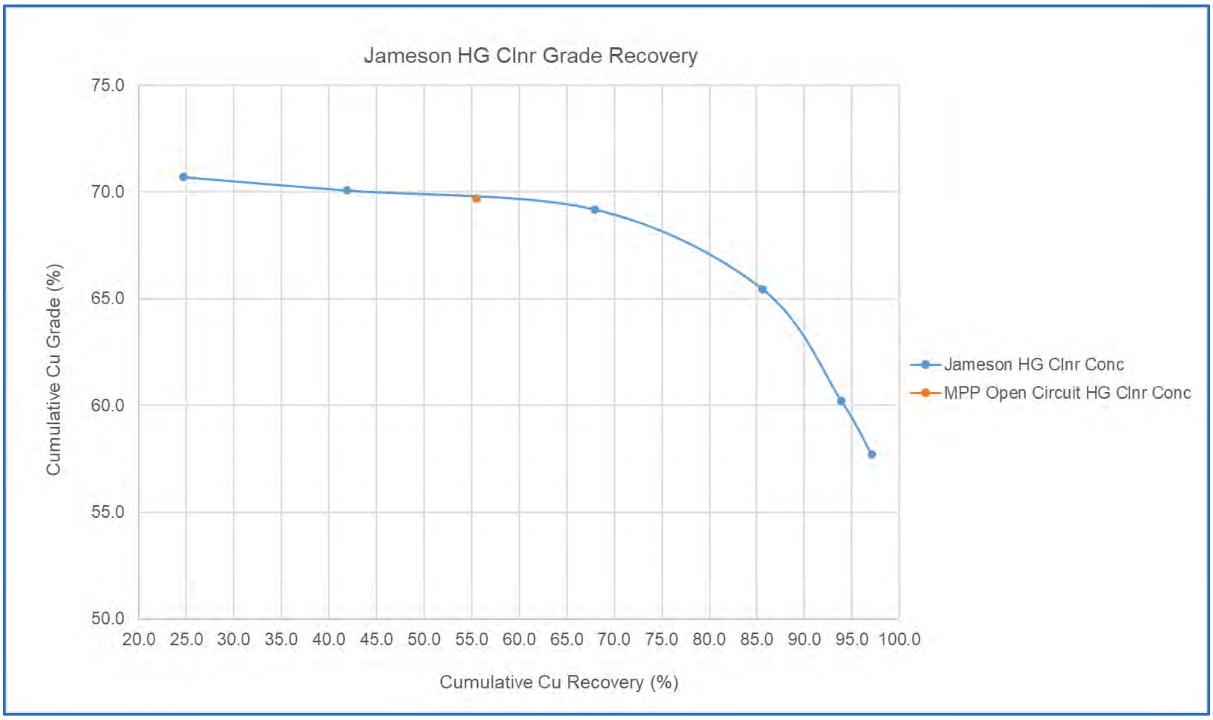
A final MPP run was conducted to produce high-grade rougher concentrate sample for Jameson cell testing, utilising a truncated flow sheet of the mainstream flotation circuit only.

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

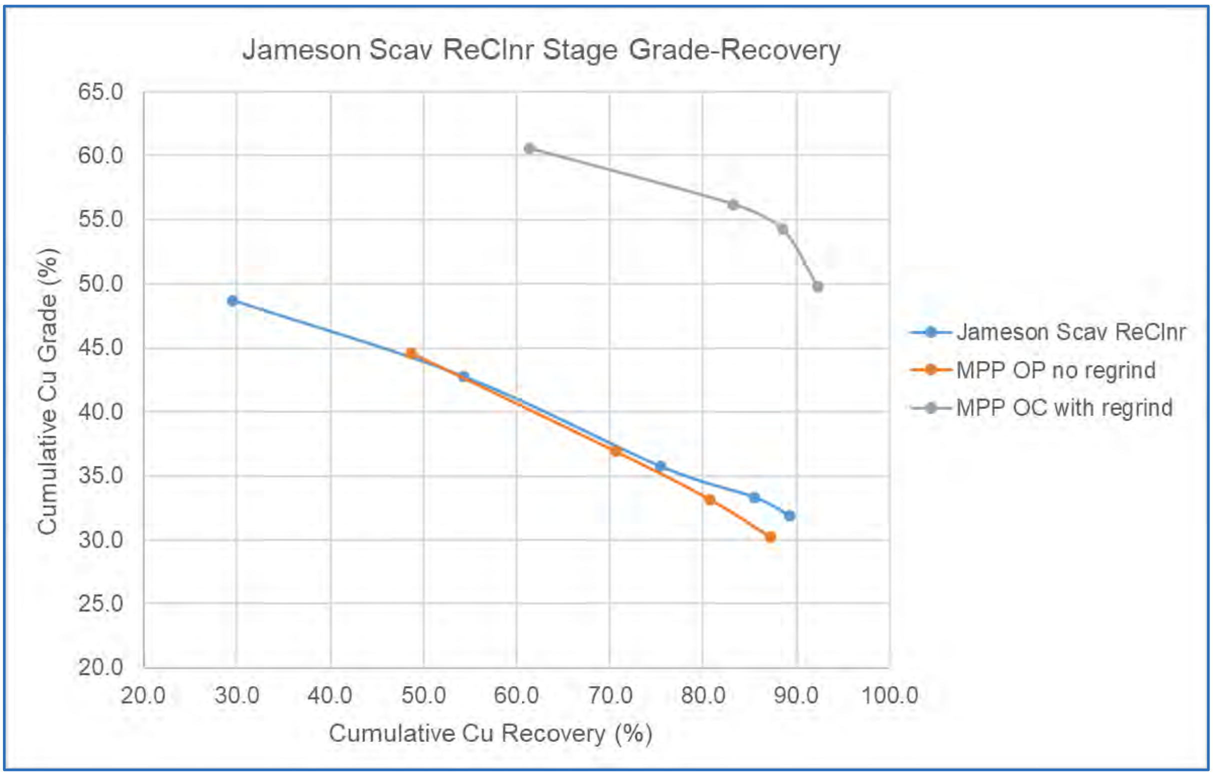


XPS, 2019.

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



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/m² 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.

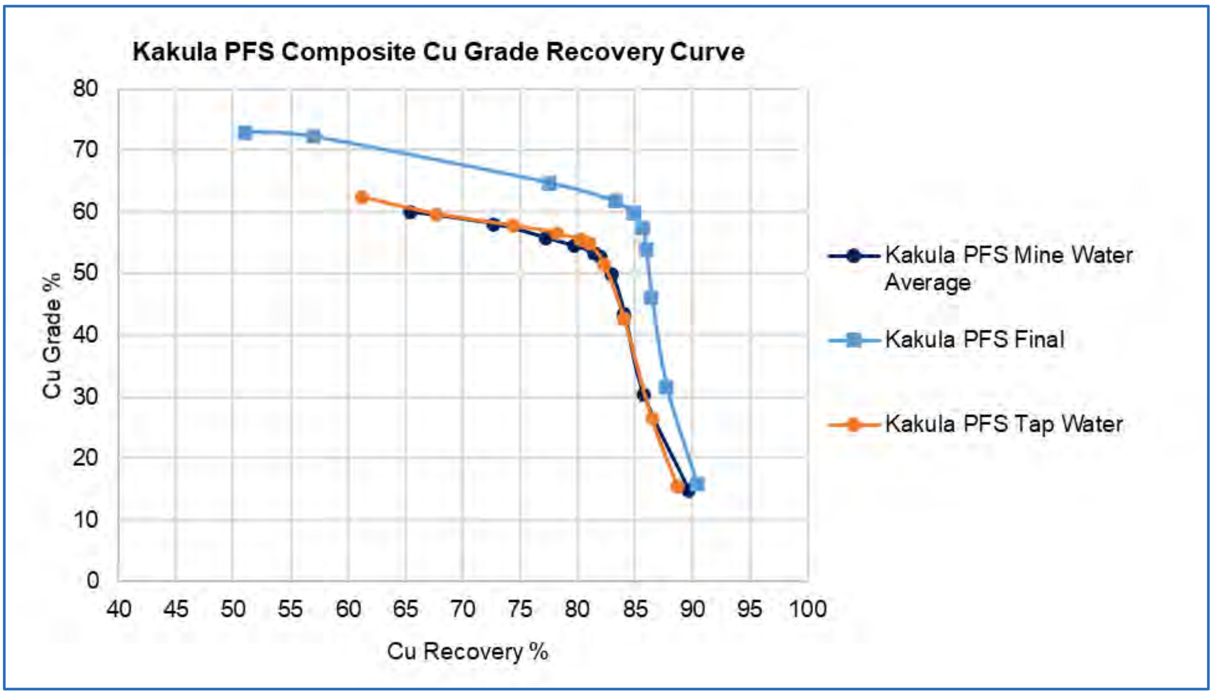
Table 13.37 Flotation Results using Mine Water

Sampel	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



XPS, 2019.

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

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 analyses 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

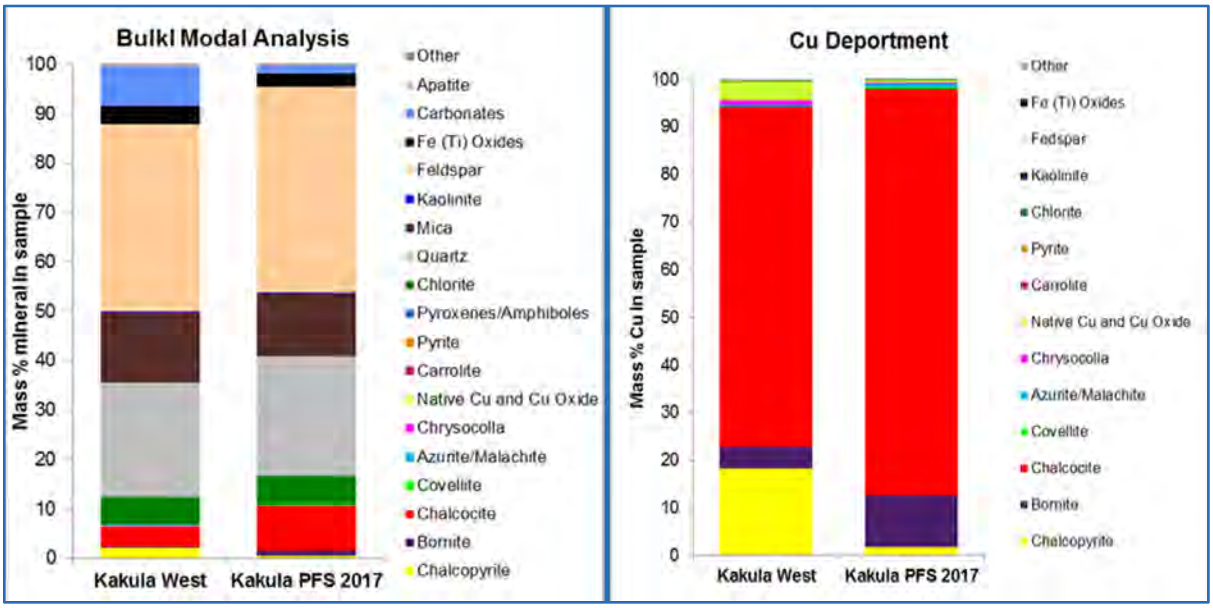
Sample	Cu (%)	S (%)	SiO_2 (%)	Fe (%)	Al_2O_3 (%)	CaO (%)	MgO (%)	As (%)
Kakula West Flotation composite sample	3.17	1.07	54.00	4.99	12.90	4.62	4.50	<0.01

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 13.44.

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.

Figure 13.44 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 Kamoa-Kakula 2023 PFS Recovery Estimate

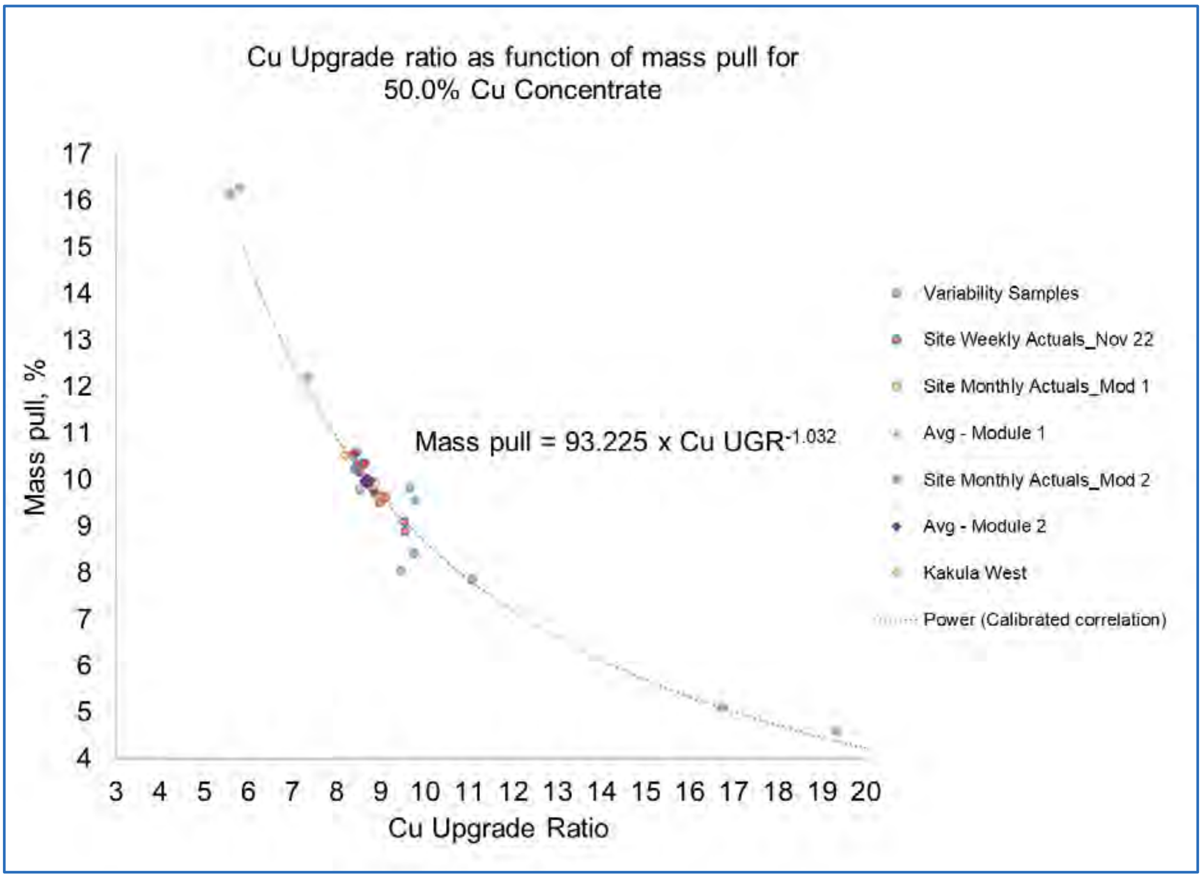
13.6.1 Kakula

The Kamoa-Kakula 2023 PFS recovery estimate for Kakula material is based on the test information generated by the Kakula Phase 1 PFS campaign, Kakula variability campaign, and current steady state operating data.

The Kakula recovery model targets a final product grade of 50.0% Cu in line with current operations. 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 (50.0%) by the individual back calculated head grades from each of the tests, and the associated mass pulls noted.

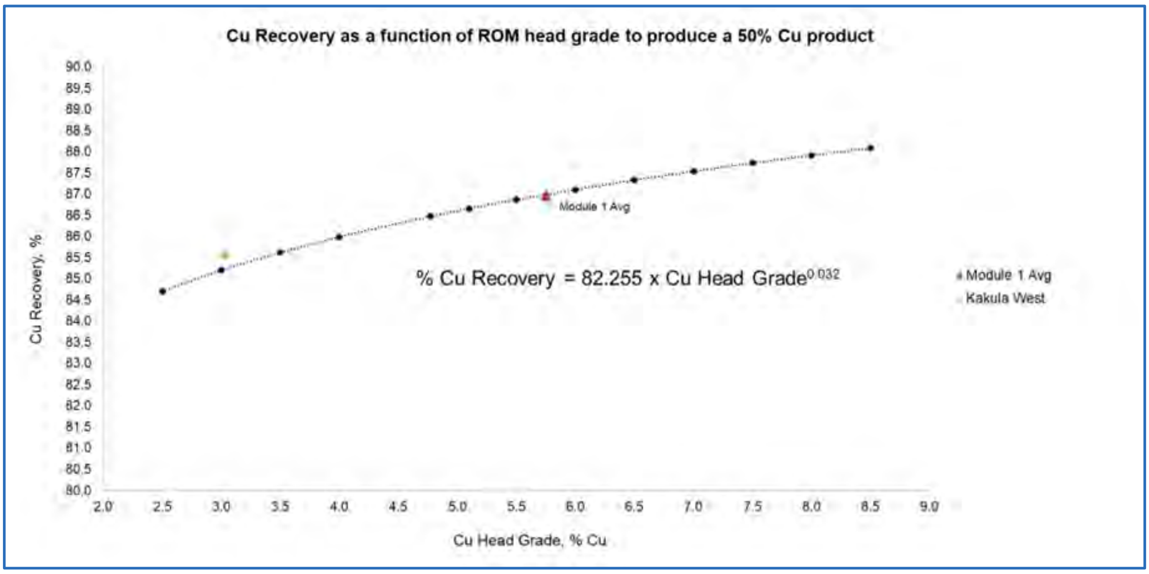
Current, steady state operating data from the Phase 1 concentrator module and Phase concentrator modules were compared to the model produced and a calibration factor was applied to ensure the correlation matched the averaged Phase 1 operational data when producing a 50.0% Cu product. The Kakula West testwork datapoint was further plotted to assess applicability. The data is presented in Figure 13.45.

Figure 13.45 Kakula Copper Upgrade Ratio versus Mass Pull



The resulting correlation from Figure 13.45 was used to calculate the expected mass pull for varying head grades, by determining the targeted Cu upgrade ratio based on a 50.0% 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 13.46.

Figure 13.46 Kakula Cu Recovery as a Function of Cu Head Grade



The mass pull and recovery correlations were applied to the Kakula Concentrator Kamoa-Kakula 2023 PFS life-of-mine production plan. No recovery discounts were applied to the Kakula estimate.

The average Cu recovery over life-of-mine to produce a 50.0% Cu concentrate was calculated at 86.3% from a 4.42% Cu head grade. The Kakula life-of-mine recovery summary is presented in Figure 13.46.

13.6.2 Kamoa

The recovery methodology discussed in Section 13.2.5.3 were applied to the Kamoa Concentrator Kamoa-Kakula 2023 PFS life-of-mine production plan. Recovery discounts were applied to the first two-years of production (2% in Year 1 and 1% in Year 2).

The average Cu recovery over life-of-mine to produce a 37.0% Cu concentrate was calculated at 87.0% from a 3.49% Cu head grade. The Kamoa life-of-mine recovery summary is presented in Figure 13.48.

Figure 13.47 Kakula Concentrator Life-of-Mine Recovery for Kamoa-Kakula 2023 PFS

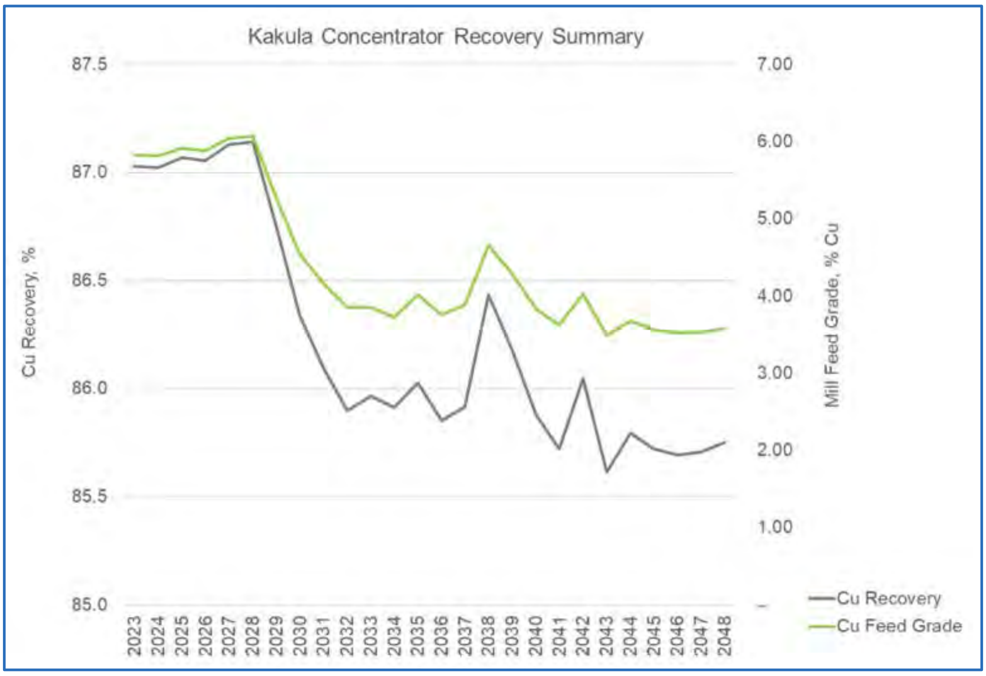
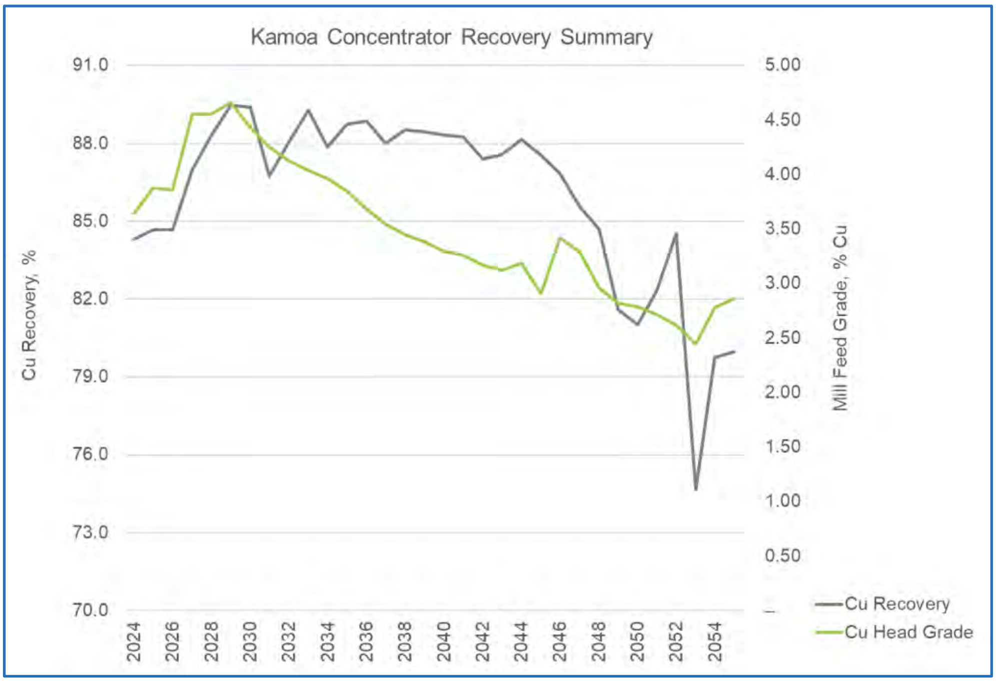


Figure 13.48 Kamoa Concentrator Life-of-Mine Recovery for Kamoa-Kakula 2023 PFS



13.7 Comments on Section 13

In the opinion of the QP the metallurgical testwork conducted for the Kamoa and Kakula deposits is sufficient for pre-feasibility 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. It further matches current operating data. 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 / threshold of 1% TCu (locally 0.5% TCu). A minimum 3 m vertical thickness was required for reporting the Mineral Resource to reflect the minimum underground mining height at Kamoa and Kakula.

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 three domains based on whether the host is the pal green siltstone, basal mineralised 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 de-surveyed drillhole file for each deposit area.

14.2 Selective Mineralised Zones

14.2.1 Kamoa

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 Ng1.1.1.1, 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 (Ng1.1.1.3) 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 Ng1.1.1 and KPS have onlapped onto the R4.2, allowing the Ng1.1.3 direct contact with the R4.2. A separate Ng1.1.3 mineralised zone was 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

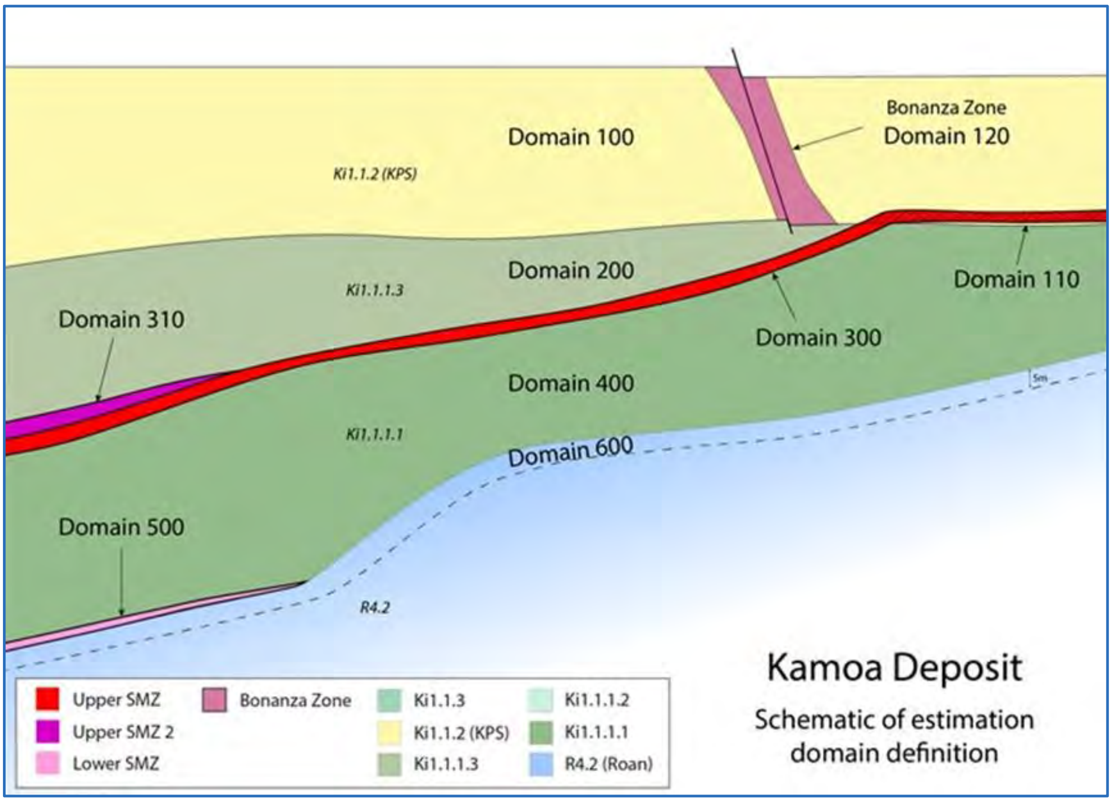
At Kakula, the highest copper grades are located just above the Roan contact in the basal siltstone. Grades usually drop sharply in the overlying diamictite, and in general, decrease gradually with increasing elevation. For resource estimation, the mineralised zone was defined using an approximate 1% 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 were modelled (Figure 14.1). These were applied to 1 m composite drillholes and the block model. Contacts between domains were 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)



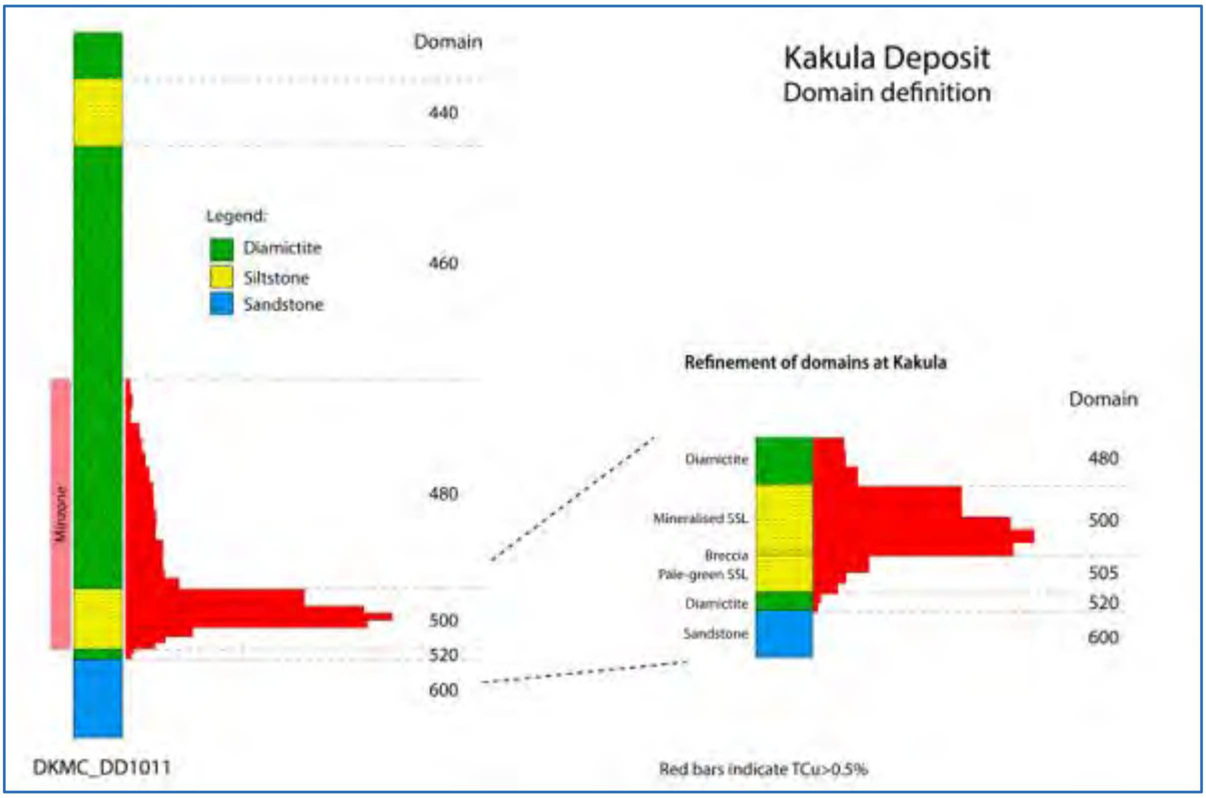
Ivanhoe, 2020.

14.3.2 Kakula

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

In addition to the seven domains, three lateral sub-domains were established to adjust the anisotropy of the search ellipse used for resource estimation, and to allow different variogram models to be used. The orientation of the search ranges were adjusted locally using dynamic anisotropy.

Figure 14.2 Kakula: Vertical Domain Definition



Ivanhoe, 2023. Refinement of domains in latest model shown on the right

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

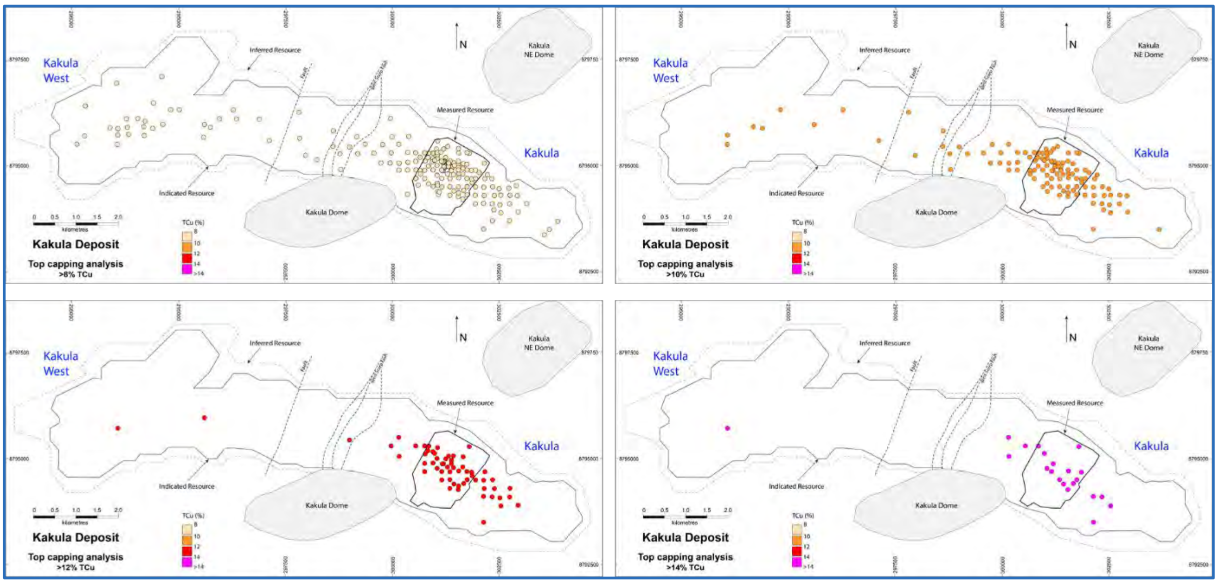
Domain	Number of Samples	Capping Grade TCu (%)	Samples Capped	No Capping		With Capping	
				Mean (%)	CV	Mean (%)	CV
100	19,927	3.0%	12	0.05	3.86	0.05	2.51
110	653	15.0%	4	2.51	0.94	2.50	0.92
120	609	35.0%	6	7.90	0.86	7.87	0.85
200	8,623	2.6%	14	0.22	1.44	0.22	1.42
300	4,627	18.0%	6	2.70	0.82	2.69	0.81
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 exist, 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 110° trend in the central portion of the deposit and along a 070° 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 0.8. TCu variograms have a low relative nugget effect (10%) and long ranges (2,000 m or longer) along the 115°, 110° and general 070° 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%



Ivanhoe, 2023.

Table 14.2 Kakula: Impact of Top Capping Per Domain on 1 m Composite Samples

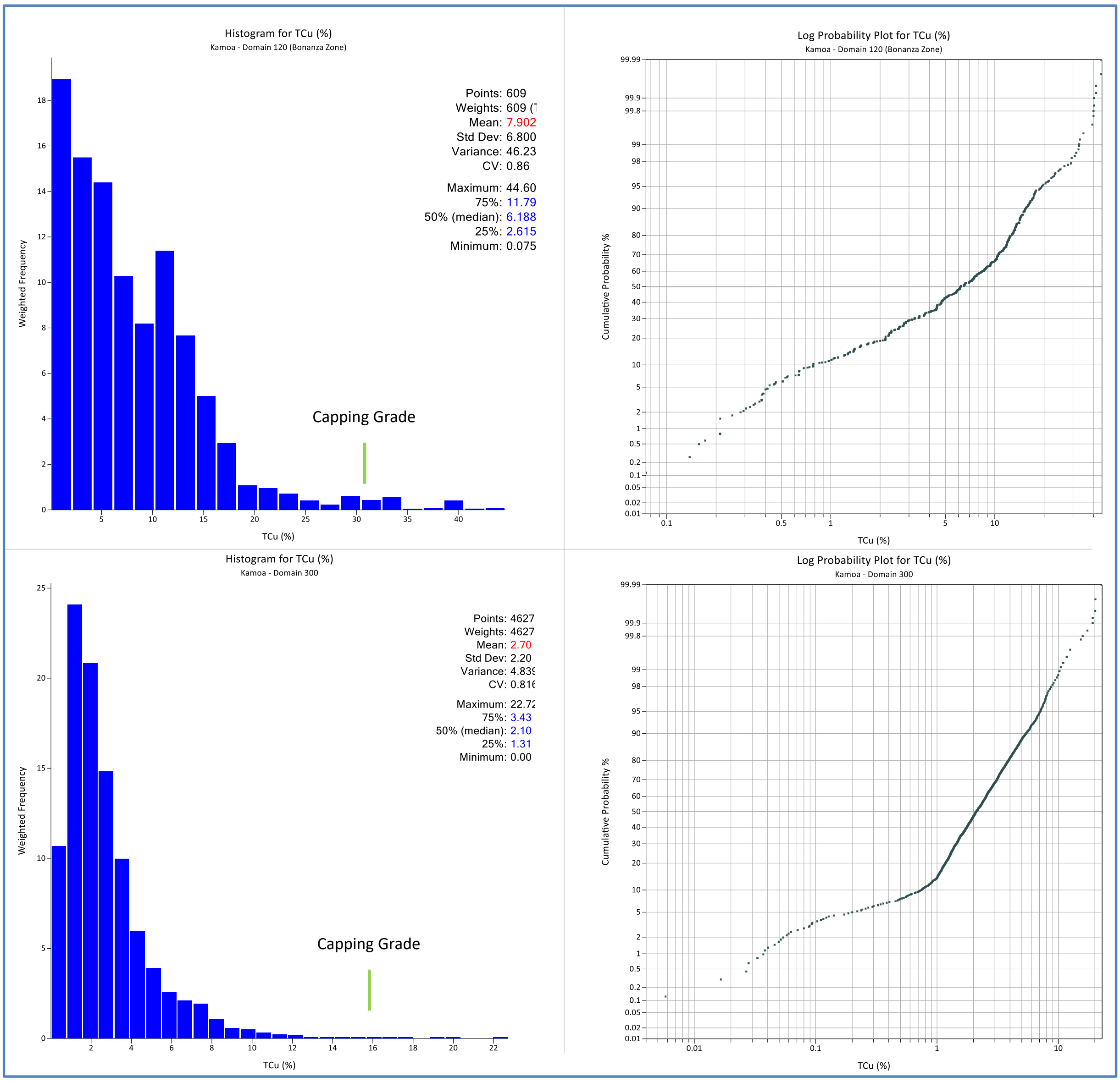
Domain	Number of Samples	Capping Grade TCu (%)	Samples Capped	No Capping		With Capping	
				Mean (%)	CV	Mean (%)	CV
460	11,288	3.0	5	0.24	1.35	0.24	1.32
480	4,612	8.5	6	1.73	0.71	1.72	0.70
505	316	5.5	4	2.06	0.54	2.04	0.50
520	1,742	3.5	15	0.60	1.20	0.57	0.95
600	2,418	3.0	13	0.14	5.08	0.12	2.82

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)



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.

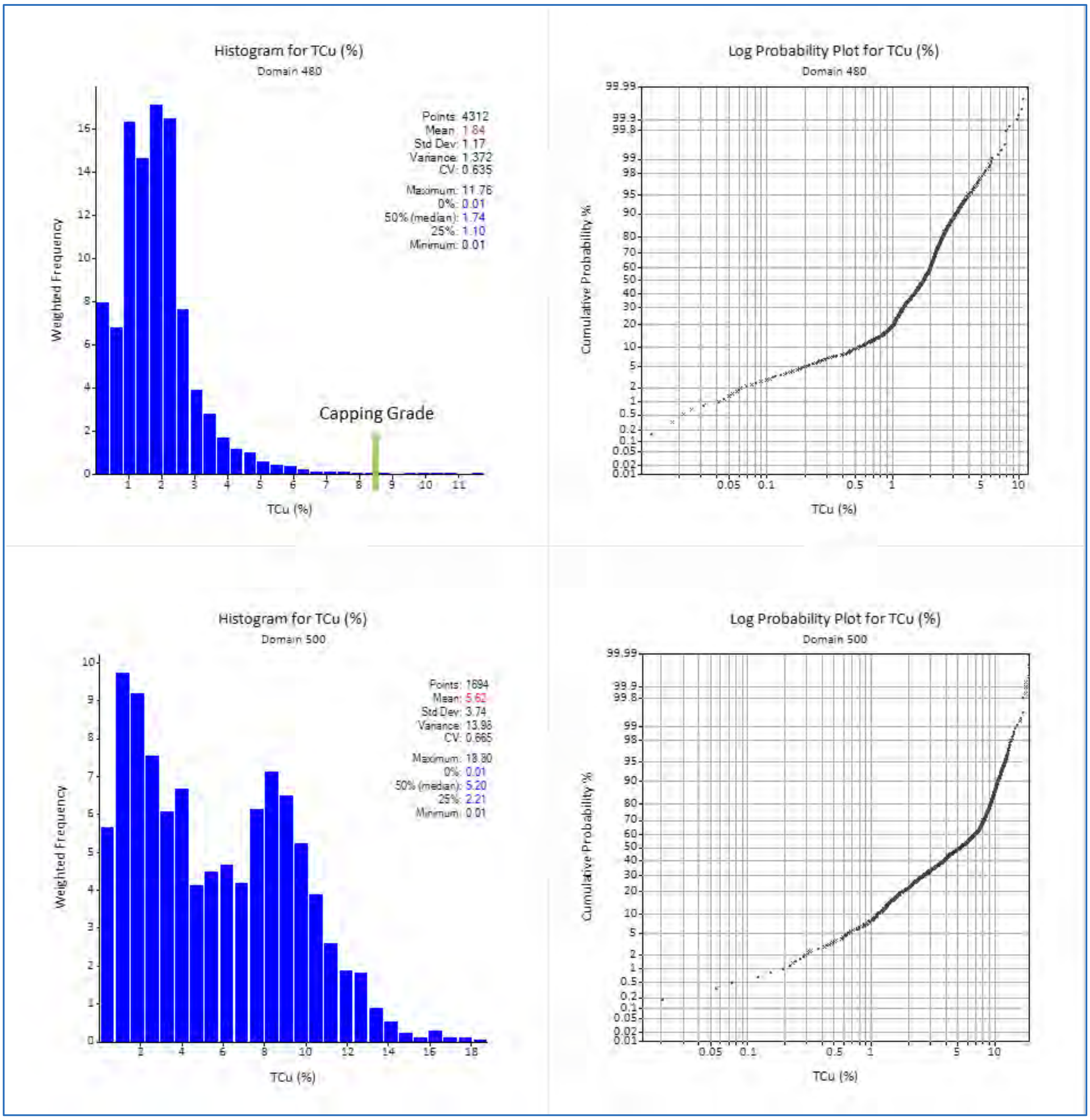
No clear relationship is evident between TCu and ASCu. Higher ASCu grades are usually highly localised and concentrated in only one or two drillholes, indicating an inability to distinguish sulfide and oxide mineralisation into separate domains. Geological and metallurgical studies of the sulfide species 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 laterally (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 portions of the basal siltstone (Domain 500). Histogram and Probability Plot



Ivanhoe, 2023.

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 estimation, 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. Mine development to date has not indicated the presence of additional brittle faults, but rather the remarkably preserved detail of the syndepositional rift geometry. Changes in elevation across these rift geometries have been successfully negotiated in mining and their discontinuous nature has been tested through underground exploration and surface infill drilling.

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

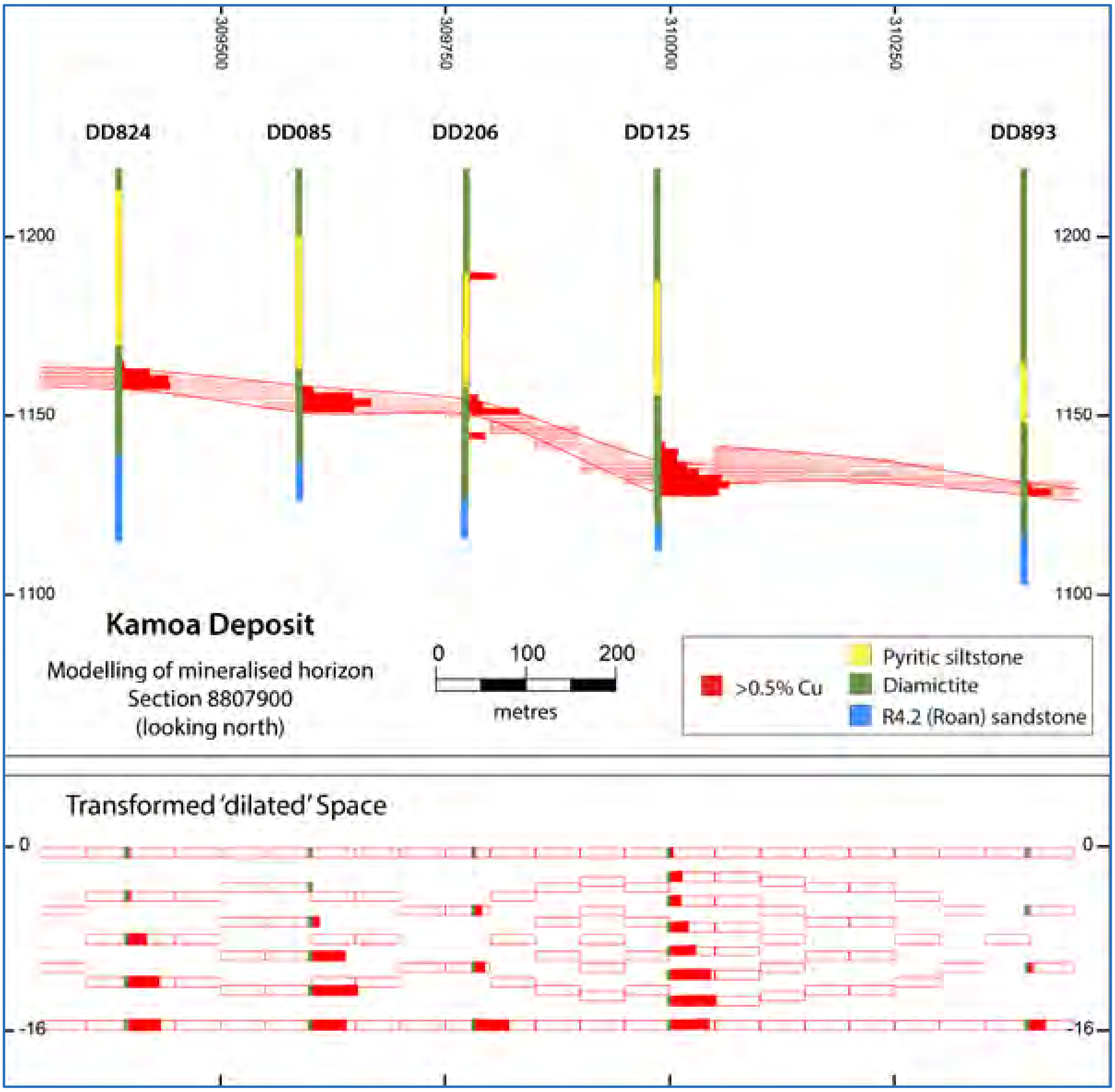
14.8.1 Kamoa

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.

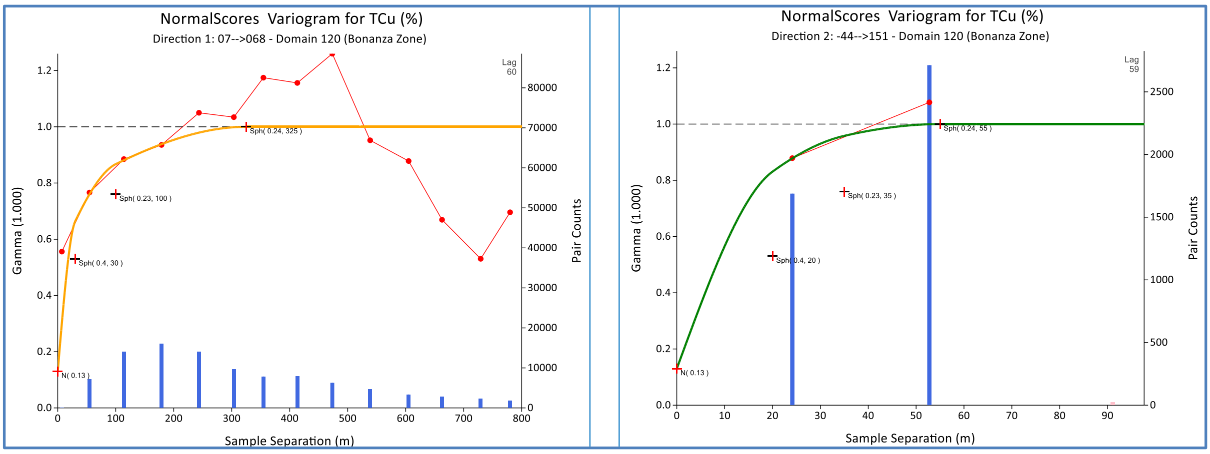
Transformed 1 m composites were used for variography. The variogram parameters were first optimised by performing sensitivity studies on the lag, angular tolerance, bandwidth and a normal-score transform prior to modelling of the variogram. The vertical bandwidth was a key parameter to preserve the vertical TCu grade profiles in drillholes and was typically set to a narrow interval. Down-hole variograms of the transformed 1 m composites were used to determine the nugget (C0). The transformation expands down-hole samples, moving them further away from each other and potentially overstating continuity at short ranges. As a validation, down-hole 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



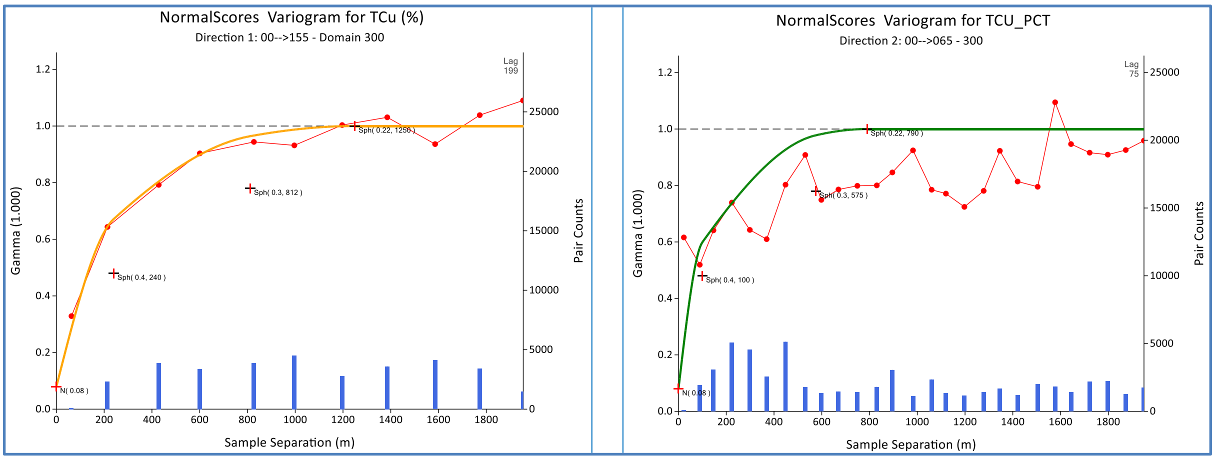
Ivanhoe, 2019; Copper grade intensity shown by bars on right side of hole.

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



Ivanhoe, 2020.

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



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 were 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

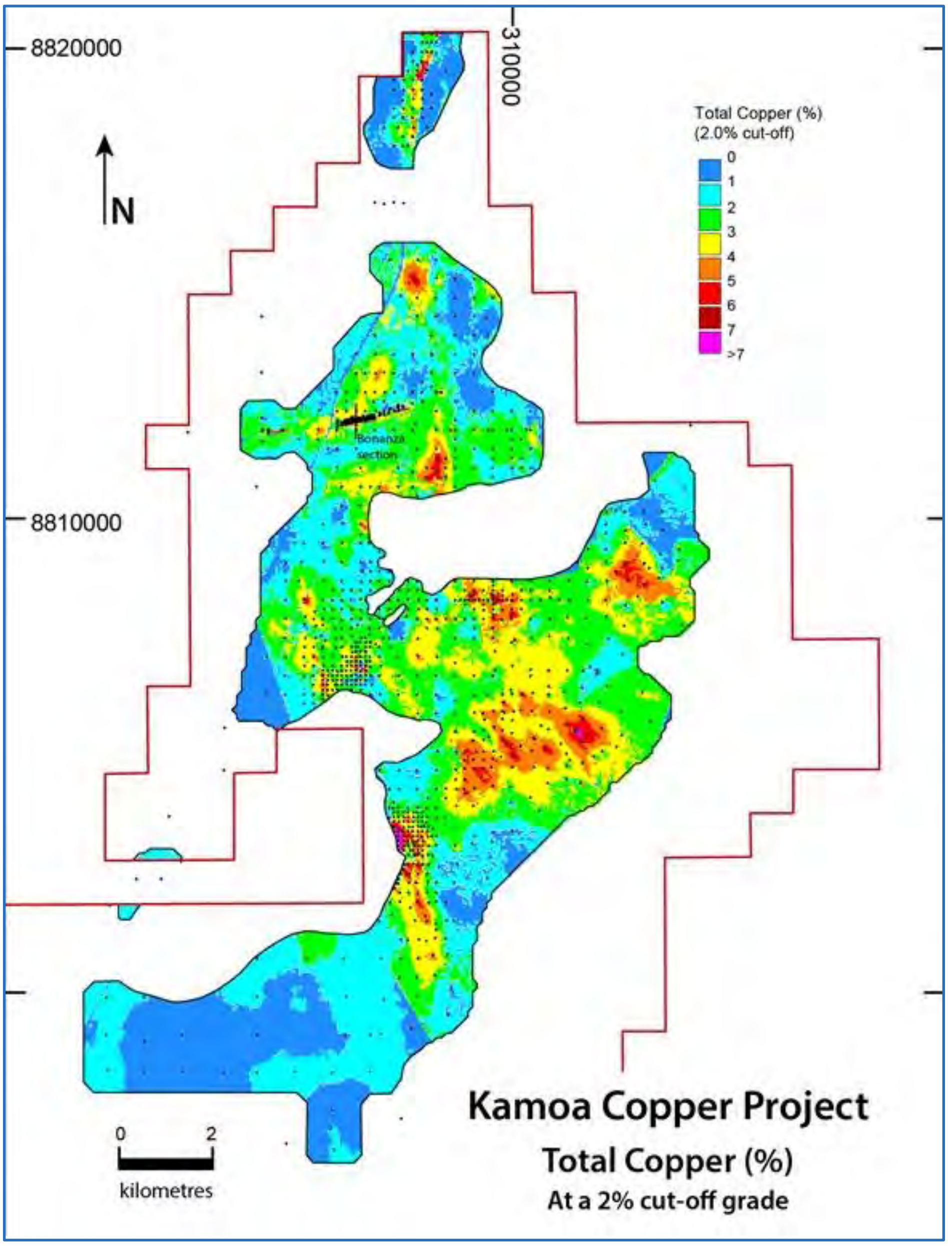
Note: Orientations shown are for overall variography; trend; these trends will vary locally as they follow the variable search orientations based on dynamic anisotropy.

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.

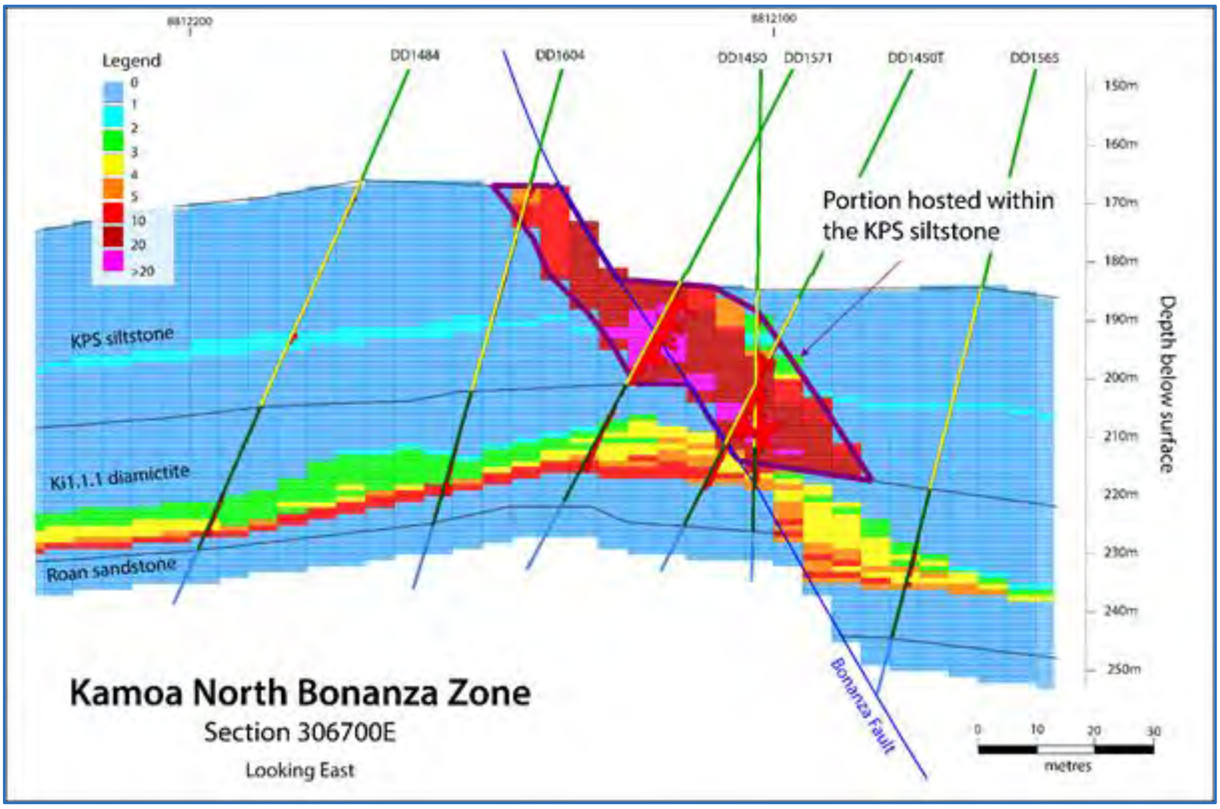
Estimated TCu grades for Kamoa are shown in Figure 14.9. A section view through the Bonanza Zone with estimated TCu grades is shown in Figure 14.10.

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



Ivanhoe, 2020. Image shows the average grade of each vertical stack of blocks above a 2% TCu Minimum 3 m thickness applied, therefore blocks below the 2% TCu cut-off grade are shown for grade trend illustration purposes.

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



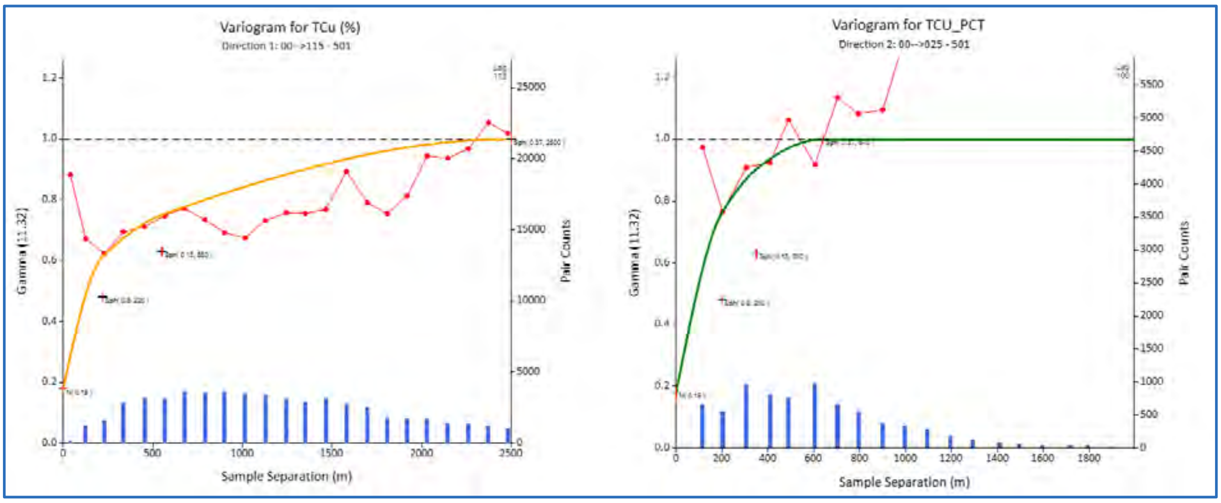
Ivanhoe, 2020.

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)



Ivanhoe, 2023.

Table 14.4 Kakula: Estimation Parameters Used for the First Search (Domain 500)

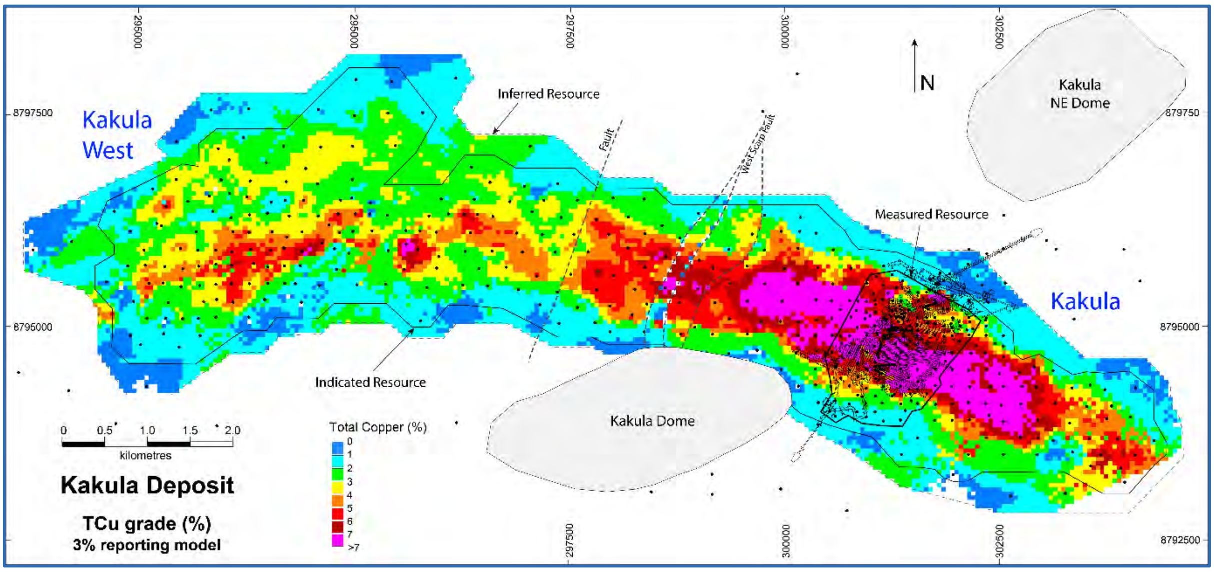
Search Domain	Orientation			Search Range	Number of Samples		Estimation Method
	Axis	Azimuth	Dip		Minimum	Maximum	
South-East	X	115°	0°	1,600	4	12	OK
	Y	25°	0°	400	4	12	OK
	Z	0°	-90°	5	4	12	OK
Central	X	110°	0°	1,100	4	12	OK
	Y	20°	0°	600	4	12	OK
	Z	0°	-90°	7	4	12	OK
Western	X	070°	0°	1,500	4	12	OK
	Y	160°	0°	400	4	12	OK
	Z	0°	-90°	6	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. Dynamic anisotropic searches were aligned along recognised controlling rift structures identified based on changes in lithological thickness of various units (Table 14.4).

Estimated TCu grades for Kakula are shown in Figure 14.12.

Figure 14.12 Plan View of Estimated TCu Grades at Kakula



Ivanhoe, 2023. Existing underground development as at December 2022. Image shows the average grade of each vertical stack of blocks above a 3% TCu cut-off. Minimum 3 m thickness applied, therefore blocks below the 3% TCu cut-off grade are shown for grade trend illustration purposes.

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 were considered in determining the Mineral Resource classification, including:

- Mapping and close-spaced sampling of underground exposures.
- 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.

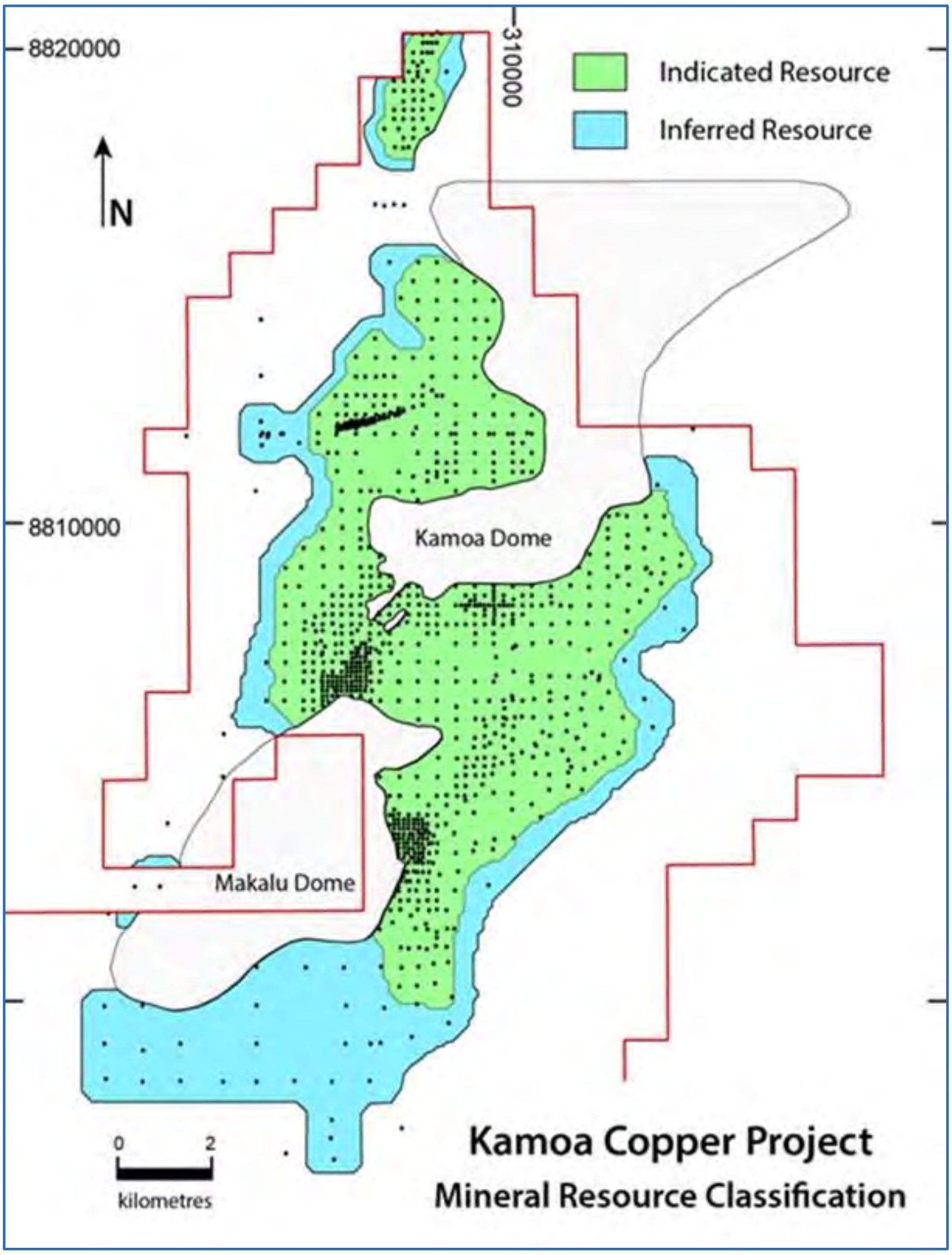
- Predictability of stratigraphic thicknesses, elevation and grade for new drillholes based on existing models.
- Modelled continuity of mineralisation and robustness of variograms for different domains; modelled continuity ranges far exceed current drillhole spacings used for classification.
- Comparison 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 27.3 km^2. Mineral Resources within a combined area of 76.9 km^2 that were drilled on 400 m spacing and which display grade and geological continuity were classified as Indicated Mineral Resources. Mineral Resources at Kakula with an area of 2.2 km^2 that were drilled on a 100-200 m spacing combined with underground exposures were classified as Measured Mineral Resources. The total area of the Kamoa-Kakula Project is approximately 397.4 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.

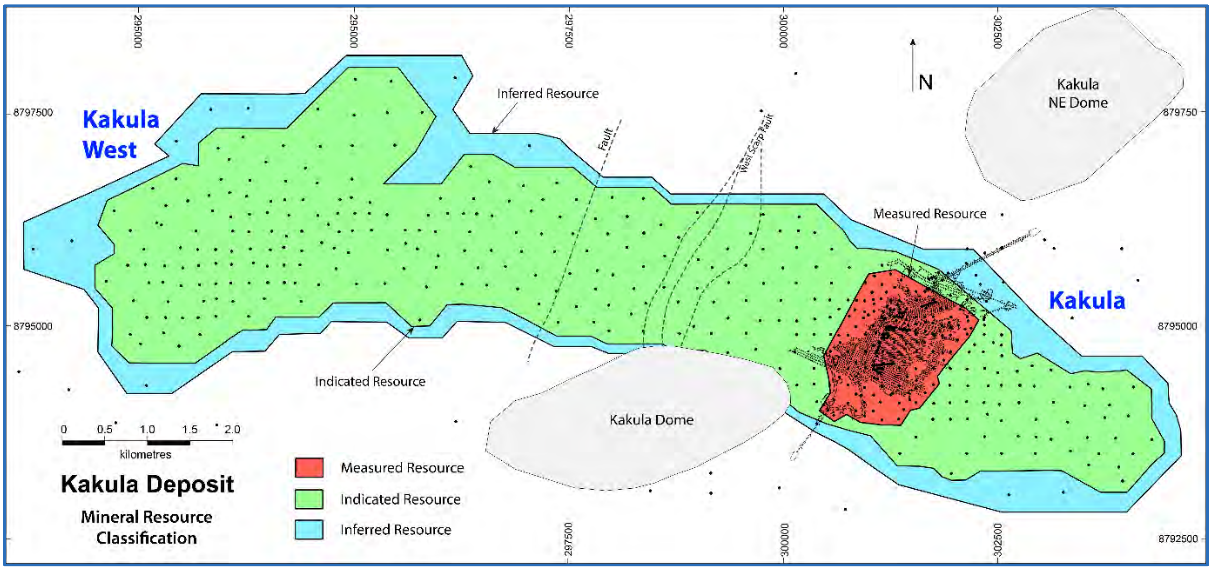
The Mineral Resource classification with drillholes for Kamoa is shown in Figure 14.13, and for Kakula in Figure 14.14.

Figure 14.13 Kamoa: Mineral Resource Classification



Ivanhoe, 2020.

Figure 14.14 Kakula: Mineral Resource Classification



Ivanhoe, 2023. Existing underground development as at December 2022.

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 good agreement.
- 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 north-west–south-east (along the trend of the high-grade mineralisation), and 500 m swaths aligned south-west–north-east (across the trend of the high-grade mineralisation). No local biases were evident.

14.12 Reasonable Prospects of Eventual Economic Extraction

MSA 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, MSA completed a cut-off grade assessment using reasonably assumed cost and revenue assumptions based on the PEA inputs. There are reasonable prospects for eventual economic extraction under the following assumptions: copper price $4.00/lb; employment of underground mechanised drift-and-fill mining methods; copper blister and concentrates will be produced and sold; average metallurgical recovery is 87.5%; mining costs are assumed to be $38/t; concentrator, tailings treatment, and general and administrative costs are assumed to be $15/t; smelter, refining and transport costs are assumed to be $13.5/t of ore at the cut-off grade; royalty of 3.5%, export tax of 1% and concentrate tax of $100/t NSR concentrate. Based on these assumptions, the Mineral Resources are considered to have met the requirement for reasonable prospects for eventual economic extraction.

MSA cautions that 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

MSA assessed reasonable prospects for eventual economic extraction for Kakula. There are reasonable prospects for eventual economic extraction under the following assumptions: copper price $4.00/lb; employment of underground mechanised drift-and-fill mining methods, and that copper blister and concentrates will be produced and sold; average metallurgical recovery is 85.5%; mining costs are assumed to be $38/t; concentrator, tailings treatment, and general and administrative costs are assumed to be $15/t; smelter, refining and transport costs are assumed to be $9.5/t of ore at the cut-off grade; royalty of 3.5%, export tax of 1% and concentrate tax of $100/t NSR concentrate. Based on these assumptions, the Mineral Resources are considered to have met the requirement for reasonable prospects for eventual economic extraction.

- MSA cautions that the Mineral Resources do not incorporate allowances for contact (external) dilution at the roof and floor of the deposit. This will ultimately depend on the ability of the mining operation to follow the SMZ boundaries.

14.13 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 Jeremy Witley, Pr.Sci.Nat SACNASP, FGSSA, 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 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 that were reviewed by Jeremy Witley, Pr.Sci.Nat SACNASP, FGSSA, employee of MSA, 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. No material from Kamoa has been processed; no depletions have been applied. 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 the following assumptions: copper price $4.00/lb; employment of underground mechanised drift-and-fill mining methods, and that copper blister and concentrates will be produced and sold; average metallurgical recovery is 87.5%; mining costs are assumed to be $38/t; concentrator, tailings treatment, and general and administrative costs are assumed to be $15/t; smelter, refining and transport costs are assumed to be $13.5/t of ore at the cut-off grade; royalty of 3.5%, export tax of 1% and concentrate tax of $100/t NSR concentrate.
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, 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 cut-off date for the drill data is 20 July 2022, with the assay table updated as of 13 December 2022. On 31 December 2022, the Mineral Resource was depleted to account for annual production; the Mineral Resource has an effective date of 31 December 2022.

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 (%)	Contained Copper (kt)	Contained Copper (billion lbs)
Measured	90	2.2	3.13	2,810	6.2
Indicated	540	21.7	2.65	14,300	31.6
Inferred	75	5.5	1.60	1,200	2.6

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 Jeremy Witley, Pr.Sci.Nat SACNASP, FGSSA, employee of MSA, who is the Qualified Person for the Mineral Resources. The cut-off date for the drill data is 20 July 2022, with the assay table updated as of 13 December 2022. On 31 December 2022, the Mineral Resource was depleted to account for annual production; the estimate has an effective date of 31 December 2022.
2. 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.
3. Mineral Resources are reported for Kakula 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 the following assumptions: copper price $4.00/lb; employment of underground mechanised drift-and-fill mining methods, and that copper blister and concentrates will be produced and sold; average metallurgical recovery is 85.5%; mining costs are assumed to be $38/t; concentrator, tailings treatment, and general and administrative costs are assumed to be $15/t; smelter, refining and transport costs are assumed to be $9.50/t of ore at the cut-off grade; royalty of 3.5%, export tax of 1% and concentrate tax of $100/t NSR concentrate.
4. Depth of mineralisation below the surface ranges from 143–380 m for Measured Mineral Resources, 12–1,373 m for Indicated Mineral Resources and 61–1,397 m for Inferred Mineral Resources.
5. Reported Mineral Resources contain no allowances for hanging wall or footwall contact boundary loss and dilution. No mining recovery has been applied.
6. Approximate drillhole spacings are 800 m for Inferred Mineral Resources, 400 m for Indicated Mineral Resources, and 100 m or underground exposure for Measured Mineral Resources.
7. 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	Tonnage (Mt)	Area (km^2)	Copper Grade (%)	Contained Copper (kt)	Contained Copper (billions lbs)
Kamoa	Indicated	760	55.2	2.73	20,800	45.8
	Inferred	235	21.8	1.70	4,010	8.8
Kakula	Measured	90	2.2	3.13	2,810	6.2
	Indicated	540	21.7	2.65	14,300	31.6
	Inferred	75	5.5	1.60	1,200	2.6
Total Kamoa-Kakula Project	Measured	90	2.2	3.13	2,810	6.2
	Indicated	1,300	76.9	2.70	35,100	77.4
	Inferred	310	27.3	1.68	5,210	11.5

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 that were reviewed by Jeremy Witley, Pr.Sci.Nat SACNASP, FGSSA, 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 14 December 2022, and the cut-off date for the drill data is20 July 2022, with the assay table updated as of 13 December 2022. On 31 December 2022, the Mineral Resource was depleted to account for annual production; the Mineral Resource has an effective date of 31 December 2022. 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 3m. There are reasonable prospects for eventual economic extraction under the following assumptions: copper price $4.00/lb; employment of underground mechanised drift-and-fill mining methods; copper blister and concentrates will be produced and sold; average metallurgical recovery is 87.5%; mining costs are assumed to be $38/t; concentrator, tailings treatment, and general and administrative costs are assumed to be $15/t; smelter, refining and transport costs are assumed to be $13.5/t of ore at the cut-off grade; royalty of 3.5%, export tax of 1% and concentrate tax of $100/t NSR concentrate.
3. Mineral Resources are reported for Kakula 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 the following assumptions: copper price $4.00/lb; employment of underground mechanised drift-and-fill mining methods, and that copper blister and concentrates will be produced and sold; average metallurgical recovery is 85.5%; mining costs are assumed to be $38/t; concentrator, tailings treatment, and general and administrative costs are assumed to be $15/t; smelter, refining and transport costs are assumed to be $9.5/t of ore at the cut-off grade; royalty of 3.5%, export tax of 1% and concentrate tax of $100/t NSR concentrate.
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, 400 m for Indicated Mineral Resources, and 100 m or underground exposure for Measured 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.

Mined material at Kakula has either been stockpiled or processed. The portion that has been processed cannot be exactly located back to the pre-mining position, precluding the reporting of the Kakula Mineral Resource at a range of cut-off grades.

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.

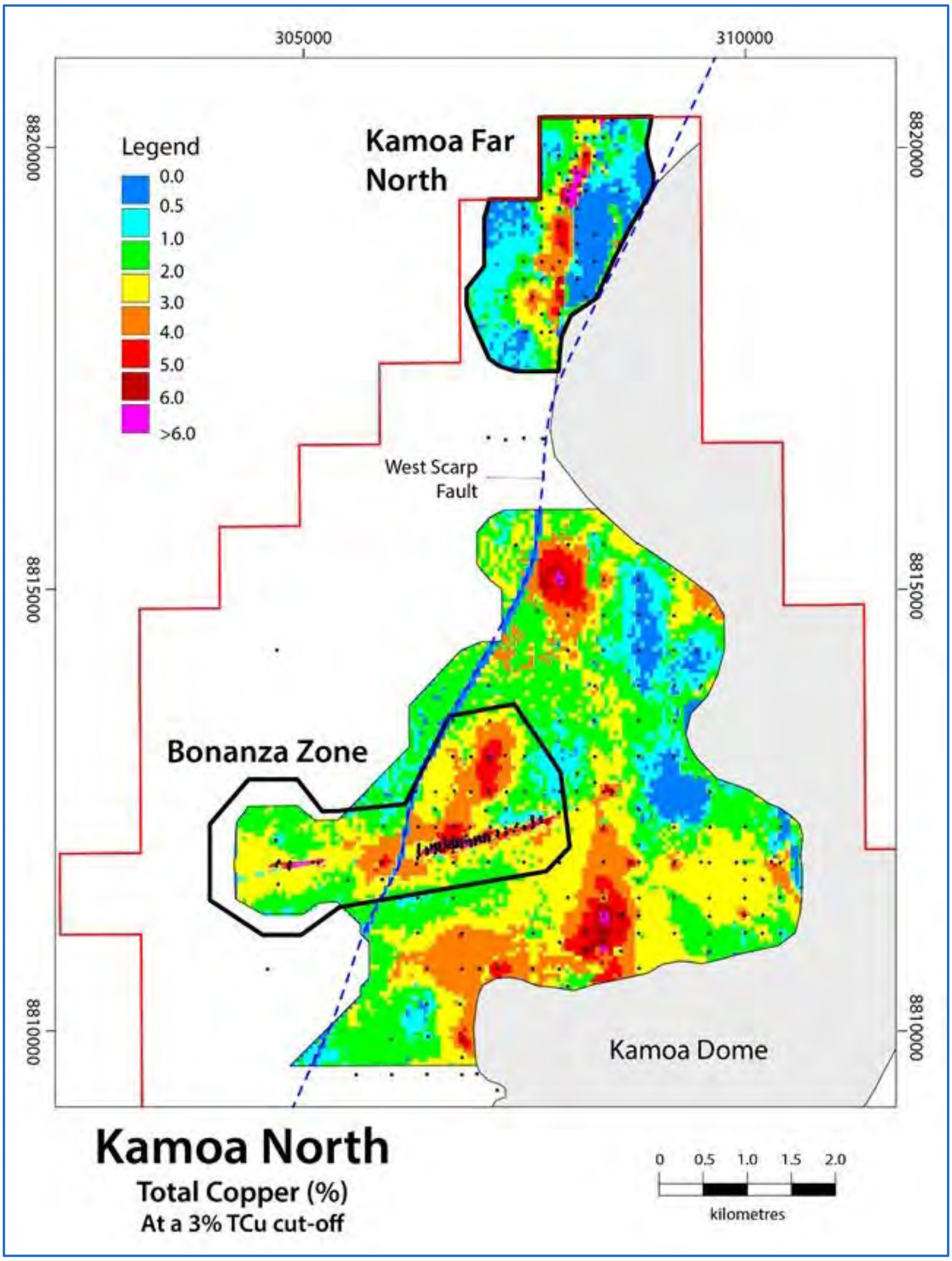
Sensitivity tables within the Kamoa North area are further divided into three areas (Figure 14.15):

- Bonanza Zone, incorporating the elevated grades within both the Ki1.1.1 (Domain 300) and KPS-hosted mineralisation (Domain 120) (Table 14.9).
- 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 14.10).

- Kamoa Far North, in the furthest northern extent of the Mineral Resource on the mining permit (Table 14.11).

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

Figure 14.15 Location of Bonanza Zone and Kamoa Far North within Kamoa North



Ivanhoe, 2020. Image shows the average grade of each vertical stack of blocks above a 3% TCu reporting cut-off. Minimum 3 m thickness applied, therefore blocks below the 3% TCu cut-off grade are shown for grade trend illustration purposes.

Table 14.9 Kamoa Bonanza Zone: 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	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, and Table 14.10 are not additive to this table.

Table 14.10 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, are not additive to this table.

Table 14.11 Kamoa Far North: 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	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, are not additive to this table.

14.15 Considerations for Mine Planning

The Kamoa and Kakula deposits pose 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 that include small scale structures which influence the mine planning, and the physical mining. 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-thicknesses 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, 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). MSA has checked the data and the methodology used to construct the resource model (Datamine macros) and has validated the resource model. MSA finds the Kamoa and Kakula resource models at Indicated or Measured classification, to be suitable to support at least pre-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, although this has not yet been the case in current underground mining operations.
  - Delineation drill programmes 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, and when negotiating growth faults.
- Assumptions used to generate the data for consideration of reasonable prospects of eventual economic extraction for the Kakula deposit.
  - Mining recovery could be lower and dilution increased where the dip locally increases on the flanks of the domes and when negotiating growth faults.

- Metallurgical recovery assumptions at Kamoa.
  - Variability testwork has been conducted on portions of the Kamoa deposit and therefore the average recoveries used in the cut-off grade.
  - Assessment may differ from actual performance. Areas of supergene mineralisation are likely to require different metallurgical parameters, however these areas make up only a small part of the deposit.

Metallurgical recovery assumptions at Kakula.

  - There is no supergene mineralisation currently identified at Kakula that requires a dedicated recovery model separate from the hypogene recovery prediction method.

- Commodity prices and exchange rates.
- Cut-off grades.

15 MINERAL RESERVE ESTIMATES

15.1 Kakula Mineral Reserve Estimate

The Kamoa-Kakula 2023 PFS Mineral Reserves have been estimated by Qualified Person Bernard Peters FAusIMM (201743), Technical Director – Mining, OreWin Pty Ltd, and Mr Curtis Smith MAusIMM(CP), Principal Mining Engineer, OreWin Pty Ltd, 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.

A Mineral Reserve has been created for each deposit within the Kamoa-Kakula 2023 PFS, which consists of Kakula, Kakula West, Kamoa 1, Kansoko Sud, and Kamoa 2. The total Mineral Reserve for the Kamoa-Kakula 2023 PFS shown in Table 15.1.

The individual Mineral Reserves can be seen in Table 15.2, Table 15.3, Table 15.4, Table 15.5, and Table 15.6. The Mineral Reserves for Kakula and Kakula West are based on the January 2020 Kakula Mineral Resource. The Mineral Reserves for Kamoa 1, Kansoko Sud, and Kamoa 2 are based on the 2020 Kamoa Mineral Resource. The effective date of the Mineral Reserve statement is 31 December 2022.

Only the Indicated portion of the Mineral Resources 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 31 December 2022. The Kamoa 2022 Mineral Resource estimate with an effective date 31 December 2022 was not completed in time to allow it to be included in the Kamoa-Kakula 2023 PFS. The Mineral Reserve defined in the Kamoa-Kakula 2023 PFS has not used all the Mineral Resources available to be converted to Mineral Reserve. The mining reserve focused on maximising the grade profile at full production rate, 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.

All production drift dilution grades are calculated by interrogating the Mineral Resource block model. Dilution due to overbreak into surrounding waste rock or backfill has been applied using factors within the schedule. All production headings that contain paste fill have a paste dilution component calculated within the schedule to produce final diluted grades. The paste fill tonnage has a zero-grade copper value. The mining recoveries includes allowances for equipment limitations, heading shapes, heading strike and dip angles, ore re-handling, and operator skill. The increase in Mineral Reserves since the 2020 Mineral Reserve in the Kamoa-Kakula Integrated Development Plan 2020 can be attributed to an increased height (7.5 m) of the second lift at Kakula, the redefining of mine boundaries at Kamoa, and the addition of Kakula West and Kamoa 2.

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 costs 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.

Table 15.1 Kamoa-Kakula 2023 PFS Mineral Reserves

Classification	Ore (Mt)	Copper (%)	Copper (Contained Mlb)	Copper (Contained kt)
Proven Mineral Reserve	–	–	–	–
Probable Mineral Reserve	472	3.94	41,055	18,622
- Kakula	138	4.79	14,580	6,613
- Kakula West	90	3.87	7,647	3,469
- Kamoa 1	121	3.74	9,963	4,519
- Kansoko Sud	38	3.70	3,088	1,401
- Kamoa 2	86	3.05	5,778	2,621
Mineral Reserve	472	3.94	41,055	18,622

1. The effective date of the Mineral Reserve statement is 31 December 2022.
2. The long-term copper price used for calculating the financial analysis is $3.70/lb. The analysis has been calculated with assumptions for an on-site smelter and excess concentrate sold to external smelters. Realisation costs include refining and treatment charges, deductions and payment terms, blister and concentrate transport, metallurgical recoveries, and royalties.
3. For mine planning, the copper price used to calculate block model Net Smelter Return (NSR) is $3.10/lb.
4. An elevated cut-off of $100.00/t NSR was used to define the stoping blocks. A marginal cut-off of $80.00/t NSR was used for development.
5. Indicated Mineral Resources were used to report Probable Mineral Reserves from the January 2020 Mineral Resource.
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 may result in apparent differences between tonnes, grade, and contained metal content.

Table 15.2 Kakula Mineral Reserves

Classification	Ore (Mt)	Copper (%)	Copper (Contained Mlb)	Copper (Contained kt)
Proven Mineral Reserve	–	–	–	–
Probable Mineral Reserve	138	4.79	14,580	6,613
Mineral Reserve	138	4.79	14,580	6,613

1. The effective date of the Mineral Reserve statement is 31 December 2022.
2. The long-term copper price used for calculating the financial analysis is $3.70/lb. The analysis has been calculated with assumptions for an on-site smelter and excess concentrate sold to external smelters. Realisation costs include refining and treatment charges, deductions and payment terms, blister and concentrate transport, metallurgical recoveries, and royalties.
3. For mine planning, the copper price used to calculate block model Net Smelter Return (NSR) is $3.10/lb.
4. An elevated cut-off of $100.00/t NSR was used to define the stoping blocks. A marginal cut-off of $80.00/t NSR was used for development.
5. Indicated Mineral Resources were used to report Probable Mineral Reserves from the January 2020 Mineral Resource.
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 may result in apparent differences between tonnes, grade, and contained metal content.

Table 15.3 Kakula West Mineral Reserves

Classification	Ore (Mt)	Copper (%)	Copper (Contained Mlb)	Copper (Contained kt)
Proven Mineral Reserve	–	–	–	–
Probable Mineral Reserve	90	3.87	7,647	3,469
Mineral Reserve	90	3.87	7,647	3,469

1. The effective date of the Mineral Reserve statement is 31 December 2022.
2. The long-term copper price used for calculating the financial analysis is $3.70/lb. The analysis has been calculated with assumptions for an on-site smelter and excess concentrate sold to external smelters. Realisation costs include refining and treatment charges, deductions and payment terms, blister and concentrate transport, metallurgical recoveries, and royalties.
3. For mine planning, the copper price used to calculate block model Net Smelter Return (NSR) is $3.10/lb.
4. An elevated cut-off of $100.00/t NSR was used to define the stoping blocks. A marginal cut-off of $80.00/t NSR was used for development.
5. Indicated Mineral Resources were used to report Probable Mineral Reserves from the January 2020 Mineral Resource.
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 may result in apparent differences between tonnes, grade, and contained metal content.

Table 15.4 Kamoa 1 Mineral Reserves

Classification	Ore (Mt)	Copper (%)	Copper (Contained Mlb)	Copper (Contained kt)
Proven Mineral Reserve	–	–	–	–
Probable Mineral Reserve	121	3.74	9,963	4,519
Mineral Reserve	121	3.74	9,963	4,519

1. The effective date of the Mineral Reserve statement is 31 December 2022.
2. The long-term copper price used for calculating the financial analysis is $3.70/lb. The analysis has been calculated with assumptions for an on-site smelter and excess concentrate sold to external smelters. Realisation costs include refining and treatment charges, deductions and payment terms, blister and concentrate transport, metallurgical recoveries, and royalties.
3. For mine planning, the copper price used to calculate block model Net Smelter Return (NSR) is $3.10/lb.
4. An elevated cut-off of $100.00/t NSR was used to define the stoping blocks. A marginal cut-off of $80.00/t NSR was used for development.
5. Indicated Mineral Resources were used to report Probable Mineral Reserves from the January 2020 Mineral Resource.
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 may result in apparent differences between tonnes, grade, and contained metal content.

Table 15.5 Kansoko Sud Mineral Reserves

Classification	Ore (Mt)	Copper (%)	Copper (Contained Mlb)	Copper (Contained kt)
Proven Mineral Reserve	–	–	–	–
Probable Mineral Reserve	38	3.70	3,088	1,401
Mineral Reserve	38	3.70	3,088	1,401

1. The effective date of the Mineral Reserve statement is 31 December 2022.
2. The long-term copper price used for calculating the financial analysis is $3.70/lb. The analysis has been calculated with assumptions for an on-site smelter and excess concentrate sold to external smelters. Realisation costs include refining and treatment charges, deductions and payment terms, blister and concentrate transport, metallurgical recoveries, and royalties.
3. For mine planning, the copper price used to calculate block model Net Smelter Return (NSR) is $3.10/lb.
4. An elevated cut-off of $100.00/t NSR was used to define the stoping blocks. A marginal cut-off of $80.00/t NSR was used for development.
5. Indicated Mineral Resources were used to report Probable Mineral Reserves from the January 2020 Mineral Resource.
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 may result in apparent differences between tonnes, grade, and contained metal content.

Table 15.6 Kamoa 2 Mineral Reserves

Classification	Ore (Mt)	Copper (%)	Copper (Contained Mlb)	Copper (Contained kt)
Proven Mineral Reserve	–	–	–	–
Probable Mineral Reserve	86	3.05	5,778	2,621
Mineral Reserve	86	3.05	5,778	2,621

1. The effective date of the Mineral Reserve statement is 31 December 2022.
2. The long-term copper price used for calculating the financial analysis is $3.70/lb. The analysis has been calculated with assumptions for an on-site smelter and excess concentrate sold to external smelters. Realisation costs include refining and treatment charges, deductions and payment terms, blister and concentrate transport, metallurgical recoveries, and royalties.
3. For mine planning, the copper price used to calculate block model Net Smelter Return (NSR) is $3.10/lb.
4. An elevated cut-off of $100.00/t NSR was used to define the stoping blocks. A marginal cut-off of $80.00/t NSR was used for development.
5. Indicated Mineral Resources were used to report Probable Mineral Reserves from the January 2020 Mineral Resource.
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 may result in apparent differences between tonnes, grade, and contained metal content.

16 MINING METHODS

16.1 Geotechnical Assessment

The geotechnical assessment for the Kamoa-Kakula 2023 PFS was undertaken by SRK Consulting (South Africa) Pty Ltd (SRK).

SRK has carried out various mining geotechnical investigations and designs for the Kamoa Copper project (Kakula, Kakula West, Kansoko, Kamoa 1 and Kamoa 2) since 2011. For the current study the following tasks were completed:

- A review on the existing Kamoa-Kakula underground geotechnical support recommendations.
- Update the geotechnical model with additional geotechnical information and provide revised support recommendations, where applicable based on Q-ratings.
- A review of the mine designs with recommendations towards the mining sequence of the planned drift-and-fill mining method (see Section 16.2), which includes sliping.

Note: The geotechnical assessment and recommendations are generic for Kakula; Kakula West; Kansoko Sud; Kamoa 1, and Kamoa 2 and will not be repeated in each of the individual sections as set out in this report.

16.1.1 Geotechnical Database

The geotechnical assessments for Kakula; Kakula West; Kansoko; Kamoa 1, and Kamoa 2 were based on the following available information:

- 2011–2013 geotechnical investigation (by SRK Vancouver).
- 2020 geotechnical investigation (SRK SA).
- Underground face mapping data collected during 2020–2022 at Kakula, and Kansoko mines.
- Feasibility level underground Geotechnical Investigation and Design for Kakula Mine. Report 541087 prepared by SRK, February 2020.
- Technical report: Kamoa-Kakula Development Plan 2020, OreWin, October 2020.

No additional geotechnical information (apart from underground face mapping) was gathered since 2020 which could be used for calculating the Q-values that serves as a guide for the underground support recommendations. As no further detailed geotechnical investigations were carried out since 2020 the structural analysis data from the Kamoa-Kakula FS 2020 were used for the current Kamoa-Kakula 2023 PFS.

16.1.2 Geotechnical Domains

The geotechnical characteristics are influenced by the three primary lithological domains, siltstone, diamictite, and sandstone (from the Roan basement rock).

16.1.2.1 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 16.1 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 16.1. The table shows the orientations of each joint set.

Figure 16.1 Kakula Structural Domains and Position of Geotechnical Boreholes

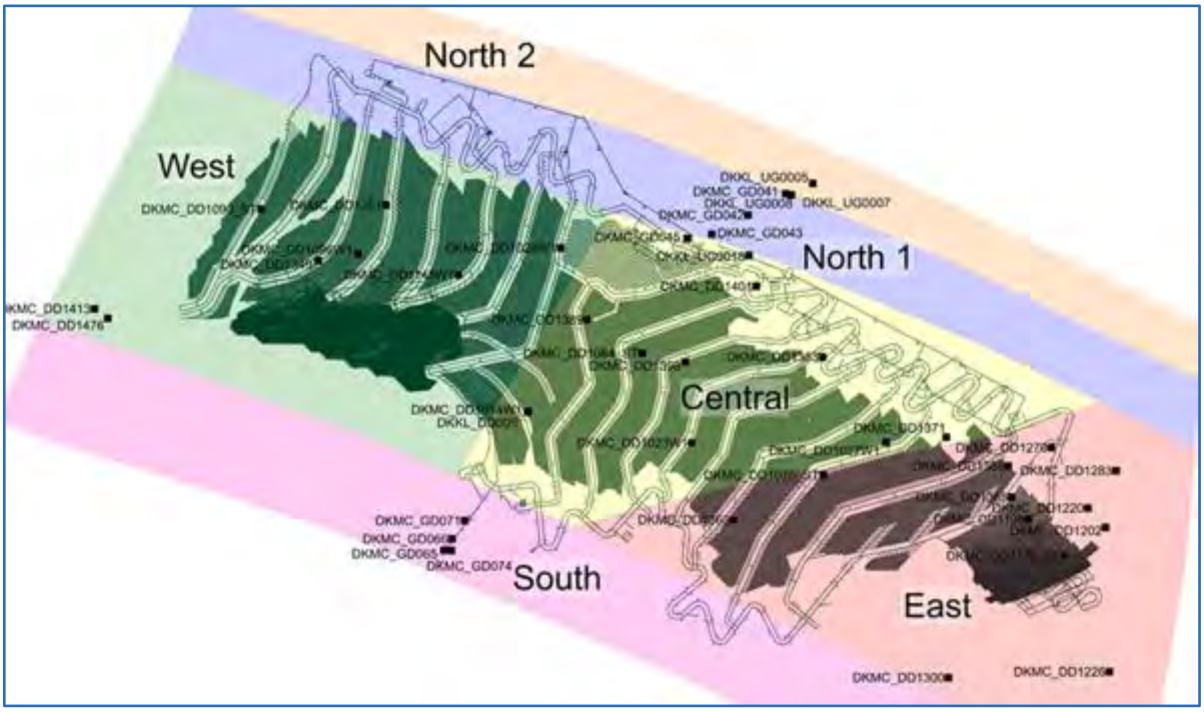


Table 16.1 Summary of Joint Sets in the Kakula Area

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

16.1.3 Geotechnical Assessment

16.1.3.1 Tunnel Quality Index (Q)

The geotechnical data obtained from the client and existing data from previous geotechnical investigations were processed and evaluated according to the Tunnel Quality Index 'Q' (Barton, 1974) rock mass classification system. This system was developed to determine rock mass characteristics and support requirements for underground tunnel excavations. The numerical value of the index Q is defined by:

The rock tunnelling quality Q is therefore considered to be a function of three parameters being crude measures of:

- Block size (RQD/Jn)
- Inter-block shear strength (Jr/Ja)
- Active stress (Jw/SRF)

According to the Q classification system there are seven rock classes ranging from extremely good (A), to exceptionally poor (G) rock conditions. Each rock class has its own set of reinforcement categories according to which support recommendations are made. Refer to Table 16.2 for the Q-rating and associated rock mass quality descriptions.

Table 16.2 Q-Ratings and Associated Rock Mass Qualities

Q-Ratings	Rock Class	Description
40.0 – 100.0	A	Very Good
10.0 – 40.0	B	Good
4.0 – 10.0	C	Fair
1.0 – 4.0	D	Poor
0.1 – 1.0	E	Very Poor
0.01 – 0.1	F	Extremely Poor
0.001 – 0.01	G	Exceptionally Poor

In order to review and recommend support strategies for the different mining areas, Q-contour maps were generated for each mining footprint. The Q contours provide an indication of the different rock mass classes expected within the direct hanging wall and at specific intervals selected above the hanging wall. For this study a 10 m wireframe interval were selected as it covers variability within the orebody and hanging wall. The evaluation is also based on median Q data since SRK is of the opinion that this is the most representative data of the geotechnical information currently available.

The geotechnical evaluation for five mining footprints 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.

SRK has reprocessed all available data collected during the 2011–2013, and 2020 geotechnical investigations as well as recently obtained underground face mapping data for Kakula and Kansoko.

The sparse geotechnical data in some areas affects the interpolation of data and generation of Q-contours, giving rise to low confidence in the spatial variability.

The poor ground conditions identified across all mining footprints are either the result of the major structures (i.e. fault zones), or due to mineralisation and oxidation in, or close to, the orebody, and in the Kamoa Pyritic Sulphide (KPS).

Groundwater inflow into the underground workings is governed by major geological structures (West Scarp Fault), and the Roan sandstone which is identified as the main aquifer in the Kamoa-Kakula mine area.

16.1.3.2 Kakula Mining Footprint

The Kakula Q-contour map illustrated in Figure 16.2 was generated from data of the previous geotechnical investigations, as well as face mapping data (484 data points). From the Q-contours, it can be deduced that the prevailing rock mass conditions are poor, fair, and good, with localised occurrences of very-poor rock mass conditions.

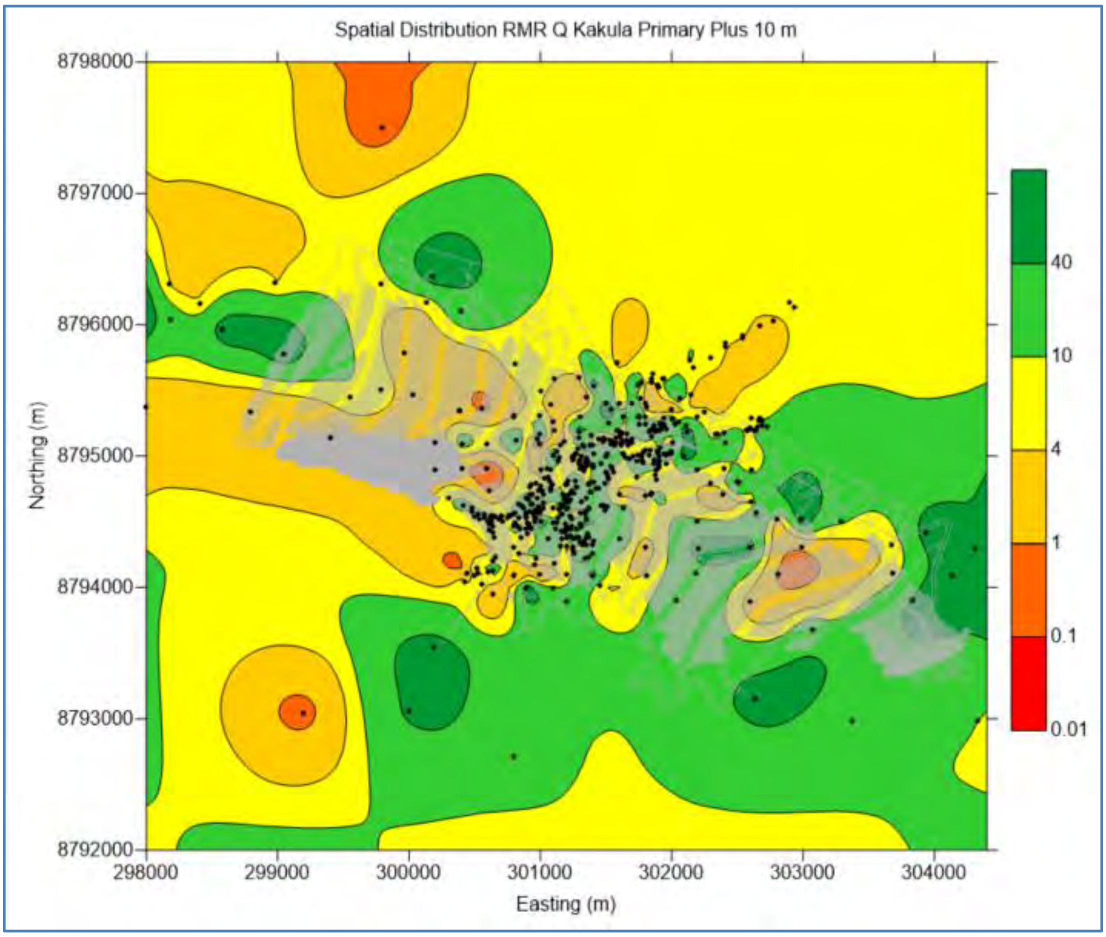
The existing geotechnical data from the boreholes were supplemented with underground face mapping data. The face mapping data supports the geotechnical data from the boreholes providing a high level of confidence that the data is representative of actual rock mass conditions within the active mining area. Kakula was also visited during March 2022, where the rock mass conditions as indicated in the Q-contour map were confirmed by visual observation.

Further away from the active mine area, only geotechnical data from boreholes is available, and the level of confidence is fair.

From Figure 16.2, it is evident that ground conditions deteriorate towards the west, which is influenced by the West Scarp Fault.

Poor, and very-poor, rock mass conditions are expected in the Kakula South steeply-dipping area and are in accordance with visual observations made from the available borehole data (Kakula South presentation by G. Gilchrist, 20220325).

Figure 16.2 Kakula Q-Contour Map



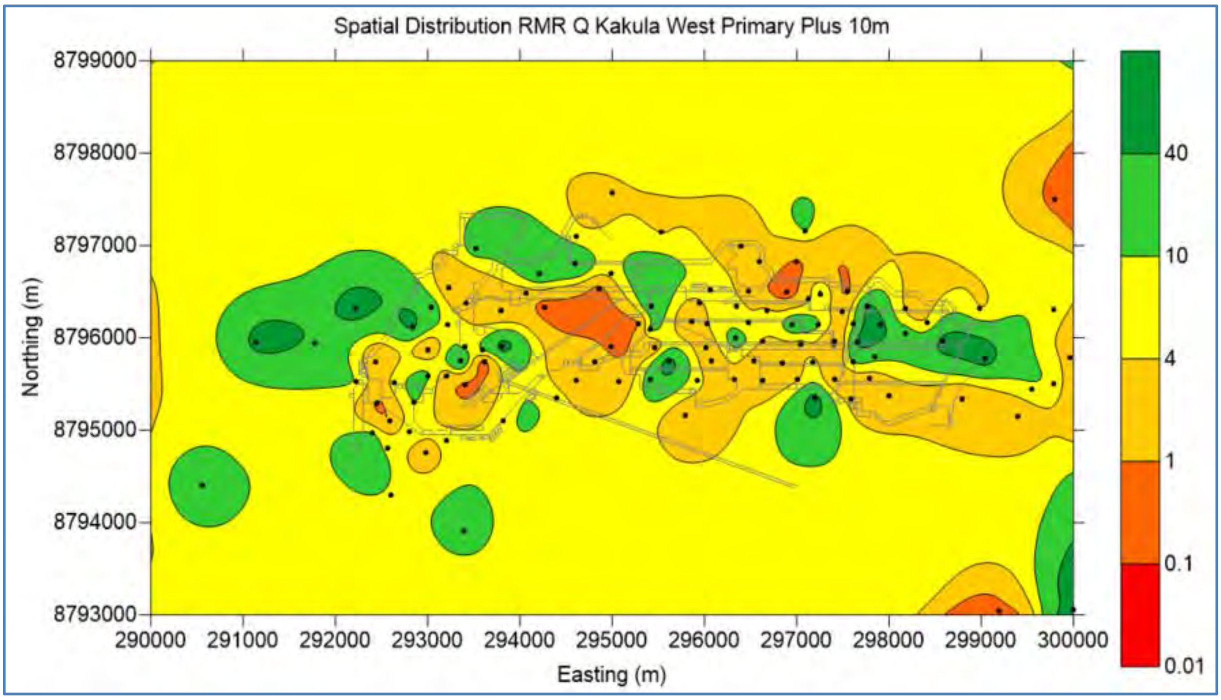
16.1.3.3 Kakula West Mining Footprint

From the Kakula West Q-contour map illustrated in Figure 16.3, it can be deduced that rock mass conditions will be poor, fair, and good, with localised occurrences of very-poor rock mass conditions.

A reasonable amount of geotechnical data (115 data points) is available for Kakula West, and boreholes are well positioned, and spaced across the mining footprint, which provides a fair level of confidence in Q-contours.

From Figure 16.3, it is evident that there are larger areas of poor, and very-poor, ground conditions when compared to Kakula, and is structurally more complex because of the effect of the West Scarp Fault.

Figure 16.3 Kakula West Q-Contour Map



16.1.3.4 Kansoko Sud Mining Footprint

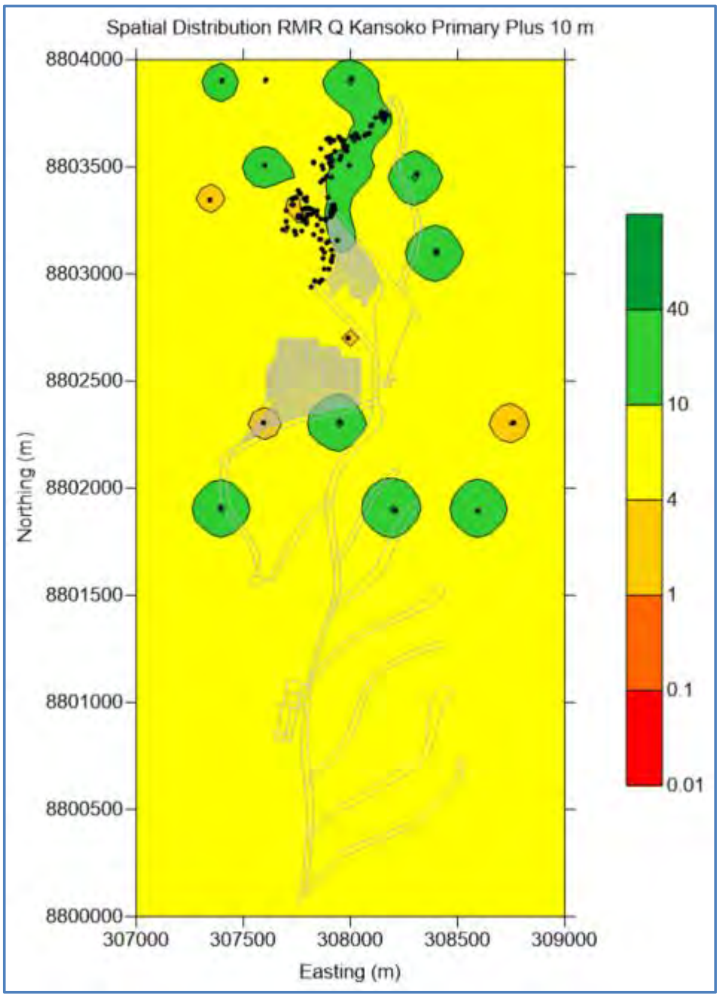
The Q-contours illustrated in Figure 16.4 indicate that the Kansoko Sud mining area has rock conditions that vary between good, fair, and poor. Localised occurrences of very-poor rock mass conditions can be expected.

The field mapping results for Kansoko confirms that rock conditions within the underground works varies between fair and good, with the presence of localised poor rock conditions. This aligns well with the earlier findings of the geotechnical investigations. During the site visit of March 2022, SRK also conducted an underground visit at Kansoko. It was noted in the drift development towards Kamoa 1, that there were localised areas of increased weathering and oxidation, and by visual observation, rock mass conditions were classified as poor.

The face mapping data supports the geotechnical data from the boreholes providing a fair level of confidence that the data is representative of actual rock mass conditions within the actively mined areas. The level of confidence does however decrease when moving further away from the active mining area as is evident in Figure 16.4

Although the geotechnical data for Kansoko is supplemented with the face mapping data (total of 145 data points), it will be required that further geotechnical drilling and logging is conducted in areas around the active mine area, in order to improve the confidence in the geotechnical model.

Figure 16.4 Kansoko Sud Q-Contour Map



16.1.3.5 Kamoa 1 and Kamoa 2 Mining Footprints

Figure 16.5 and Figure 16.6 illustrate the generated Q-contour map for Kamoa 1 and Kamoa 2 respectively, from which it can be deduced that rock mass conditions will vary between extremely poor, poor, fair, and good. It should however be noted that a limited number of boreholes (58 data points) were drilled within both mining footprints, which affects the interpolation of geotechnical data across the footprint. The 'white' areas are representative of areas where data is so limited that it is not possible for the software to interpolate the data.

Due to the spatial variability in the interpolated data SRK has a low level of confidence in the Q-contours, and it is recommended that infill drilling, and geotechnical logging is done to increase the level of confidence in the geotechnical model. Additional geotechnical information will also inform detailed geotechnical considerations towards the impact of the KPS on future mining activities. The KPS is expected to present poor ground conditions (as seen from photographs), as well as the requirement for special ground support associated with a highly corrosive environment.

Figure 16.5 Kamoa 1 Q-Contour Map

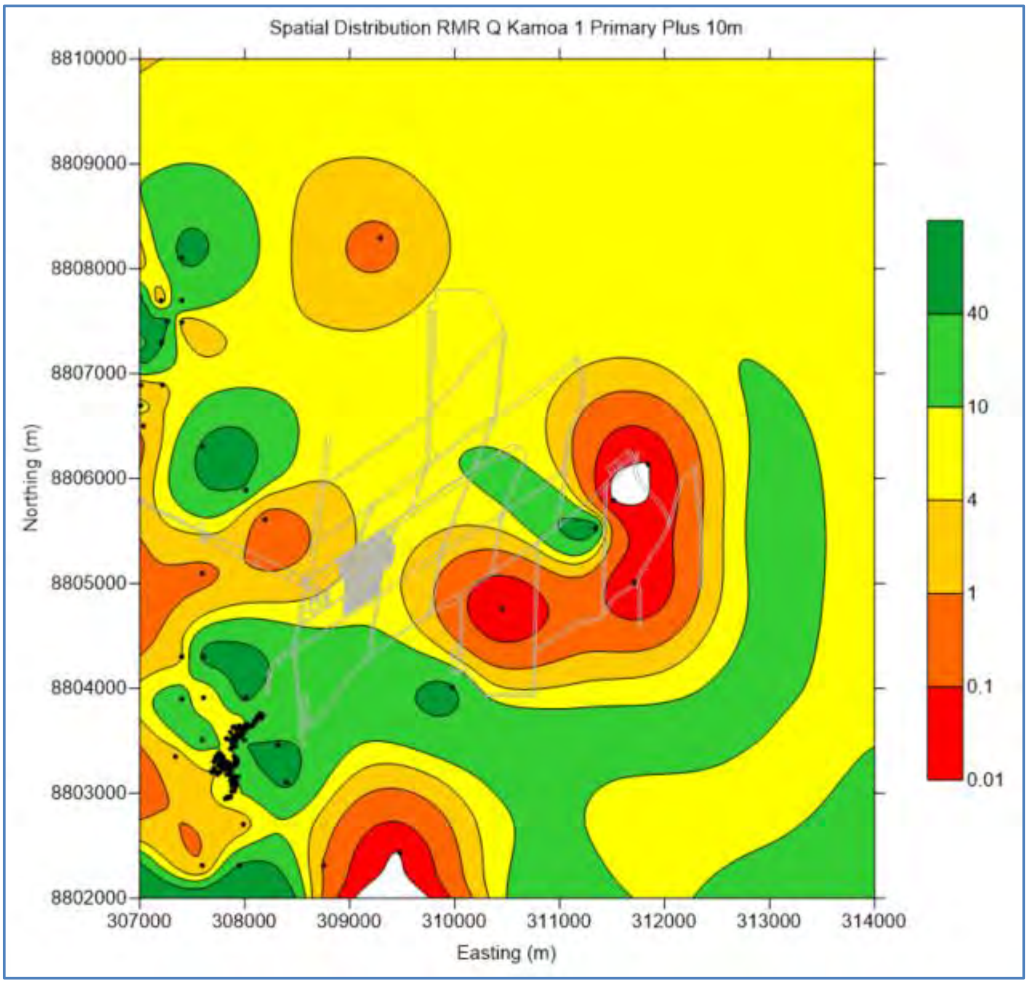
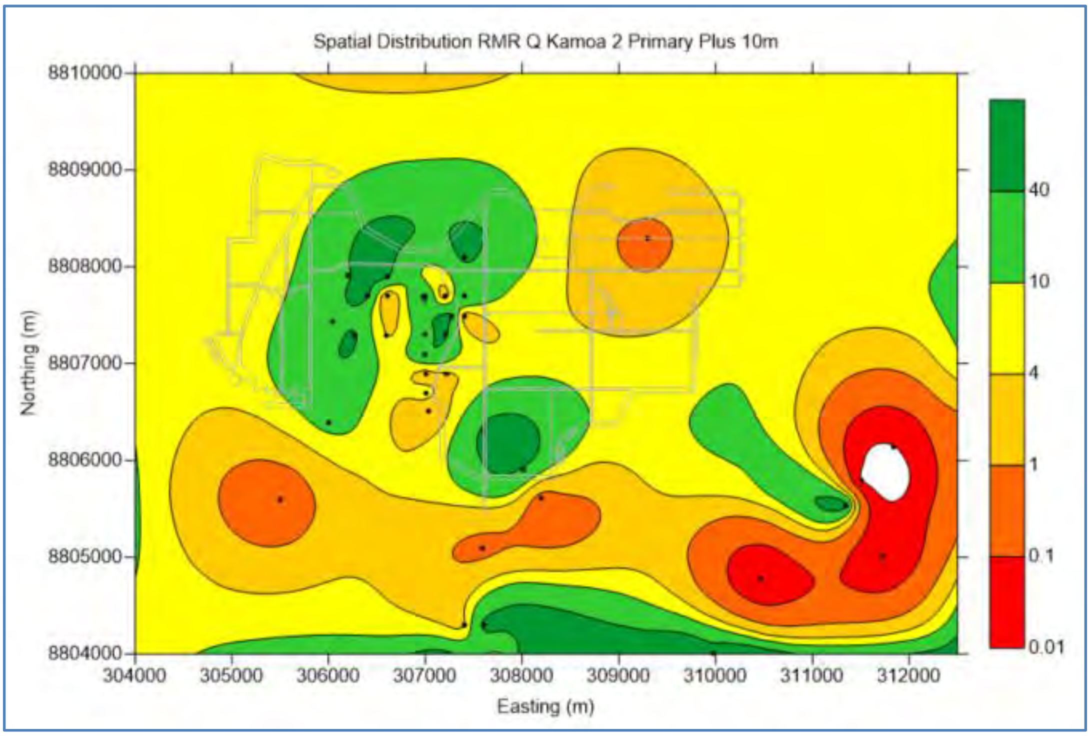


Figure 16.6 Kamoa 2 Q-Contour Map



16.1.4 Support Requirements

This geotechnical evaluation addresses the drift-and-fill with sliping mining method in detail. Since this mining method is still under trial, SRK has also included the support recommendations for conventional drift-and-fill in the instances where drift-and-fill with sliping might not be possible (i.e. poor, and very-poor rock conditions). Hanging wall access drift-and-fill (HWAD&F) support recommendations are provided for mining in steeply-dipping areas.

Since the detailed geotechnical assessments for the alternative mining methods were addressed in previous geotechnical reports, it will not be repeated in this report, and only a brief description with the detailed support recommendations for each method is included in this report.

Support recommendations are also made towards large infrastructure, twin connection drifts, access and perimeter drifts, and workshops.

16.1.4.1 Support Design Considerations

Rock Mass Quality

Primary support must cater for the local rock mass conditions. The geotechnical data obtained from the client were processed and evaluated according to the Tunnel Quality Index 'Q' (Barton, 1974) rock mass classification system. There are five rock mass quality classes (Q) identified in the Kamoa-Kakula mining footprint according to which this support evaluation was done.

The support design of the tunnels in Kakula (6–8 m wide excavations, and 7.6 m high) was based on the guidelines presented by Potvin and Hadjigeorgiou (2016) design chart (see Figure 16.7), in conjunction with the rock mass classes identified. The main design decision for tunnel support guidelines according to Potvin and Hadjigeorgiou (2016) are based on the following:

- The reinforcement pattern is expressed as a bolt density (bolts/m^2).
- The type of surface support such as mesh and/ or reinforced shotcrete.
- The thickness of shotcrete, where reinforced shotcrete is used (mesh considered as shotcrete reinforcement).
- The coverage of the ground support down the walls includes tunnels where bolts are installed in the roof only.

The comprehensive benchmarking review which informs the guidelines by Potvin and Hadjigeorgiou (2016) is based on the Ground Control Management plans implemented in mainly Australian and Canadian hard rock underground mines.

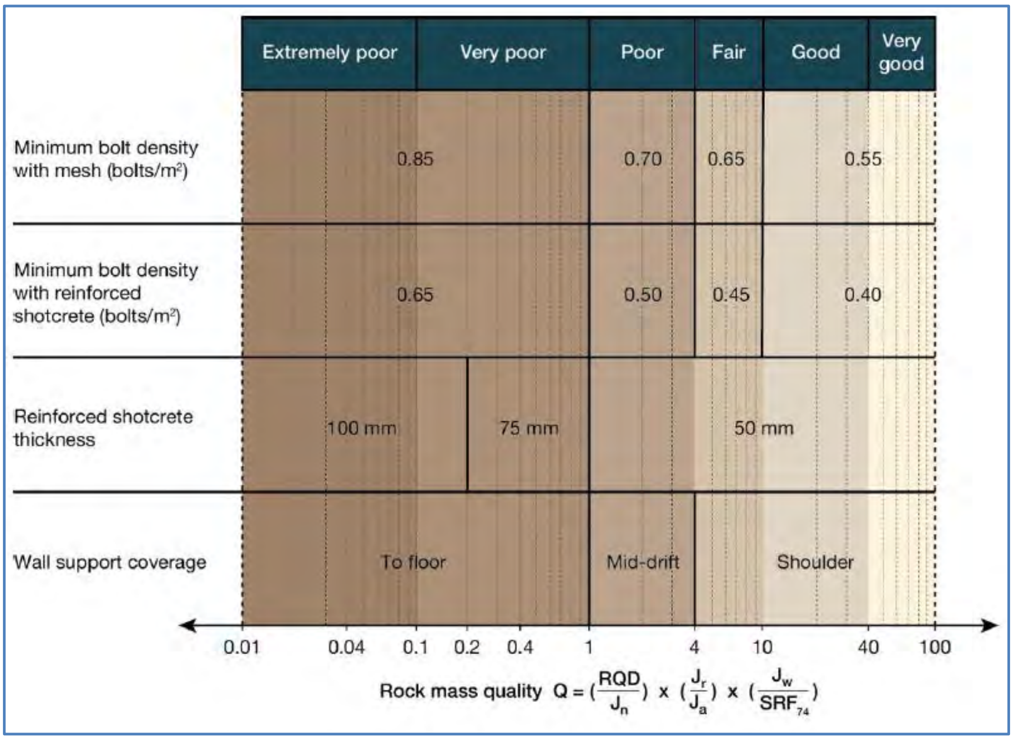
For this evaluation the relation between rock mass quality and support requirements was based on the correlation between the rock mass classification (Q') and the ground support pressure requirement.

It should be noted that the guidelines are used in conjunction with an:

- Actual understanding of the site ground conditions.
- Requirement for the excavations (permanent / temporary).
- Long-term / short-term support requirement.
- Low – or high-risk exposure.

The support recommendations by SRK have been based on the above with some being more conservative and other less conservative than presented in the guidelines.

Figure 16.7 Ground Support Guidelines (Potvin and Hadjigeorgiou)



The support pressure for different rock mass classes was calculated by means of the following equation (Barton, 2002):

$Pr = 0.1Q^{-1/3}$

This is an empirical estimate of the support pressure requirements and is compared with the support resistance provided by the resin rebar for each support class in Table 16.3.

Table 16.3 Support Pressure, and Support Resistance provided by Resin Rebar

Rock Mass Classification	Q	Support Pressure (kN/m²)	Spacing (Horizontal)	Spacing (Vertical)	Support Resistance (kN/m²)	FoS (Rebar)
Extremely poor	0.01 – 0.1	464 – 215	0.8	0.8	475	0.6 – 1.4*
Very poor	0.1 – 1.0	215 – 100	1.0	1.0	297	0.9 – 1.9*
Poor	1.0 – 4.0	100 – 63	1.0	1.0	190	1.9 – 3.0#
Fair	4.0 – 10.0	63 – 46	1.5	1.5	84	1.3 – 1.8
Good	10.0 – 40.0	46 – 29	1.5	1.5	>84	1.8 – 2.9

The FoS for the very-poor rock mass conditions are low, but the support density is higher than in Figure 16.7, where the shotcrete thickness is increased to 75 mm. It is recommended that 75 mm shotcrete is applied in addition to the rebar.

Where extremely poor ground conditions occur, more surface support will be required with greater resistance. With reference to Potvin and Hadjigeorgiou (2016), a localised, single occurrence of extremely poor rock mass conditions was supported with mesh and 100 mm shotcrete. For the purposes of this study, this support is recommended, but local circumstances may dictate more or less intense support, which will be determined when these conditions are encountered during mining.

It must be noted that where very-poor, and extremely-poor ground conditions are encountered, the development and sliping round length will need to be reduced to 2 m or less to enable support to be installed timeously and to prevent overbreak and unravelling. When enlarging the span to 14 m, it may be too difficult to manage stability reliably. In these cases, it will be necessary to backfill the secondary stope and the portion of the tertiary stope that has been sliped. Once the backfill has cured, the tertiary drift can be developed with the required support. It may be necessary to change the sequence to a typical primary, secondary, tertiary drift-and-fill sequence where these conditions are encountered.

Ground Support for Large Infrastructure

For specific large chambers or stopes /cubbies, 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. In general, for the critical infrastructure excavations where width or height is more than 9.0 m, 3.0 m long resin bar in a 1.0 m x 1.0 m pattern, with 100 mm of reinforced shotcrete, and 6.5 m long cable anchors are advised.

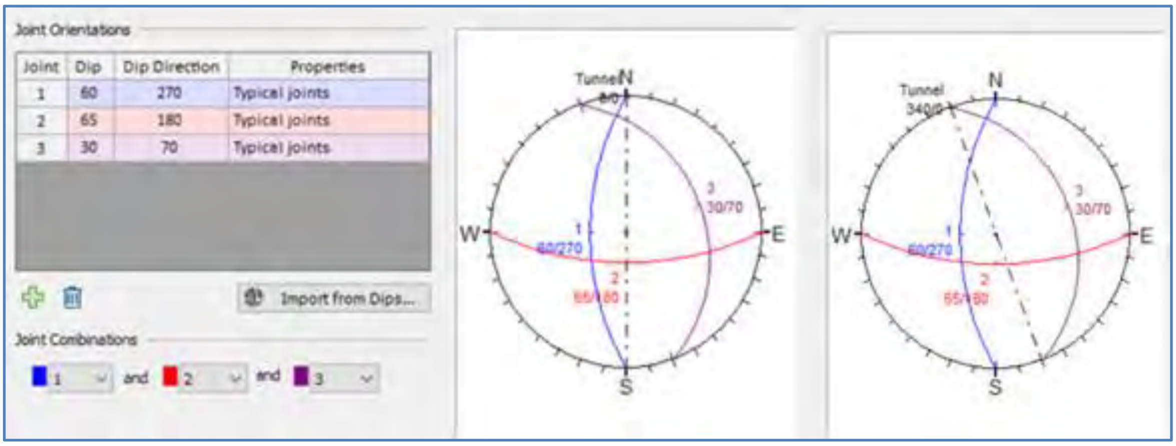
Secondary Support for Sliping (drift-and-fill method)

The secondary support is specific to drift-and-fill mining when large spans are created during sliping. When sliping takes place, the increased span (14 m) will enable the formation of larger wedges, which will need to be supported. Overbreak and poor tight filling may increase the effective span and further increase the size of potentially unstable wedges. Secondary support in the form of cable anchors is required to cater for large wedges.

A 3D Unwedge analysis was conducted by using the Rocscience software to determine the suitability of the proposed support spacing and support length.

The joint sets used for creating the wedge are based on actual joint information presented in the Kakula FS report (2020). Figure 16.8 illustrates typical joint orientations in accordance with the latest mine design for Kakula. Tunnels that are oriented north–south, north–north-west, and south–south-east are the least favourable orientations with respect to these joint sets.

Figure 16.8 Joint sets used for creating a representative wedge based on existing information.



The Unwedge analysis addresses two scenarios:

- Actual 14 m wide span with effective tight filling (illustrated in Figure 16.9).
- Equivalent 20 m wide effective span to represent the effect of overbreak and poor tight filling (illustrated in Figure 16.10 and Figure 16.11).

Results of the Unwedge Analysis for a 14 m wide span with tight fill:

Figure 16.9 illustrates the support regime in a 14 m wide span with a 5.40 m apex height. The support is done with 3 m long resin bolts at a 1.5 m x 1.5 m spacing. Although the support does not continue through the apex of the wedge the volume of 'unsupported' wedge is small enough that 3 m long resin bolts is sufficient to prevent a wedge failure at a FoS = 1.53.

Results of the Unwedge Analysis for a 20 m effective span with poor tight filling:

Figure 16.10 illustrates the support configuration in a 20 m wide effective span with a 7.72 m apex height.

The 3 m long resin bolts at 1.5 m x 1.5 m spacing do not provide sufficient support since a large volume of wedge (towards the apex) is not intersected by the bolts, resulting in a FoS<1.0.

Figure 16.11 illustrates the scenario where one of the outside anchors are angled at 60° towards the tertiary slipe.

From the results of the Unwedge analysis, it can be deduced that for a 14 m wide excavation with tight fill, and support with 3 m resin rebar support, will be adequate at a FoS >1.5. However, since there is a risk that tight fill will not be achieved due to various factors, and due to expected overbreak, it is recommended that 6 m cable anchors at a 3 m x 3 m spacing are installed.

The Unwedge analysis for a 20 m wide effective span indicates a FoS of 1.0, which is deemed acceptable, since this is an extreme case with the least favourable tunnel orientation, and poor tight filling and overbreak. There is also limited exposure after sliping and these excavations will have a limited duration.

This design enables the cable anchors to be installed during the development of the secondary drift, so that the full span is pre-supported when sliping takes place. To speed up installation, the cables can be anchored, and do not need to be grouted. Since the cable anchors are deemed a short-term requirement, corrosion is not expected to be a concern. In the longer-term, corrosion of the cable anchors will result in less complications during mining of a second cut.

Figure 16.9 Tight filling – Actual 14 m Excavation Span

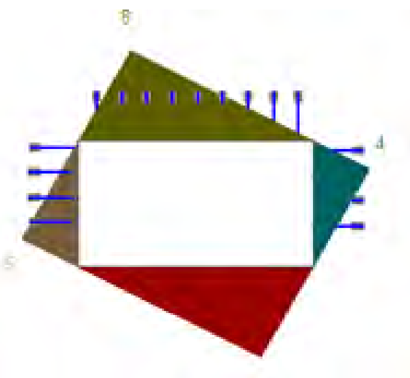
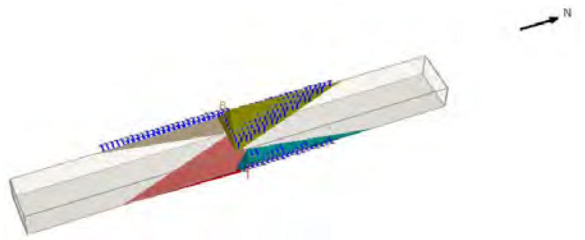
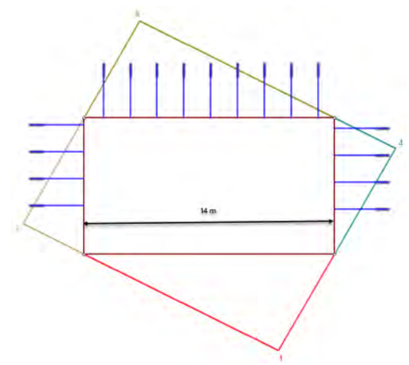
Roof wedge (8) FoS = 1.53 Weight = 13.097 MN Apex height = 5.40 m			
	16.9A	16.9B	16.9C

Figure 16.10 Poor tight filling – Equivalent 20 m Excavation Span (no cables)

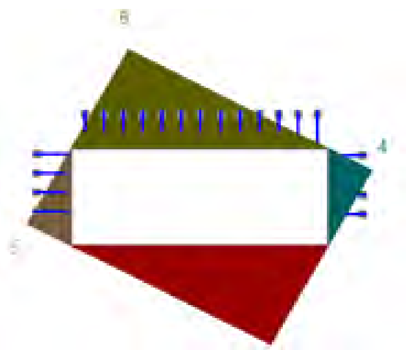
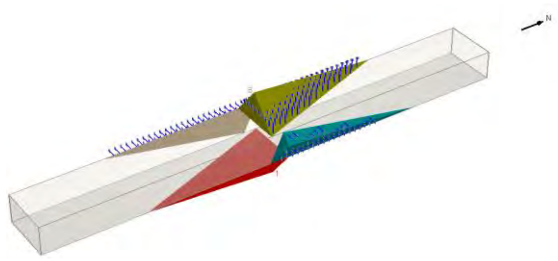
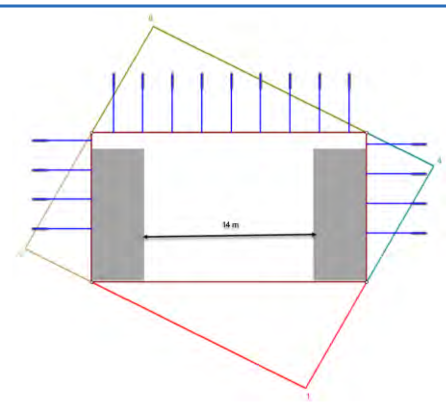
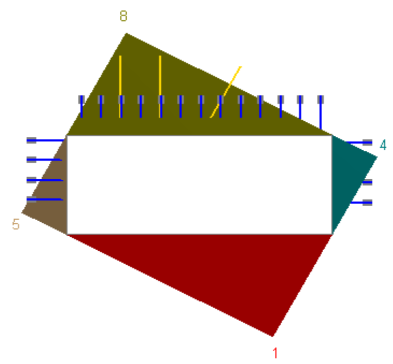
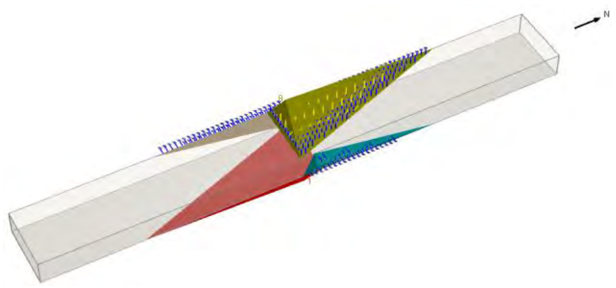
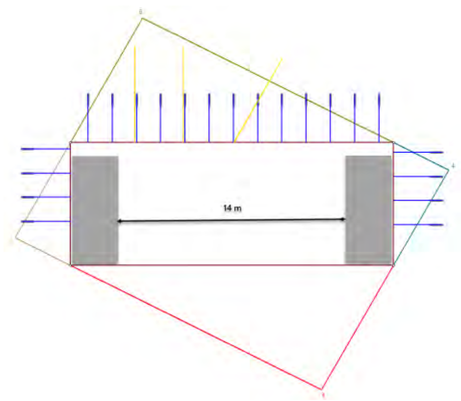
Roof wedge (8) FoS = 0.85 Weight = 38.185 MN Apex height = 7.72 m			
	16.10A	16.10B	16.10C

Figure 16.11 Poor tight filling – Equivalent 20 m excavation span (3 m x 3 m cables with 1 = 60°)

Roof wedge (8) FoS = 1.10 Weight = 38.185 MN Apex height = 7.72 m			
	16.11A	16.11B	16.11C

It must be noted that the sliping could be affected by the localised, poor rock, mass conditions. This may require that sliping is stopped, followed by backfilling of the drift-and-slipped area, and then developing the remainder of the tertiary drift. In this case, a more typical drift-and-fill method would need to be applied, to limit the unfilled span. The support pattern for primary drifts should then be applied to the secondary and tertiary drifts.

Ground Support for Hanging Wall Access Drift-and-Fill Mining Areas (steeply-dipping deposit)

Resin bolts are proposed as a primary option for ground support in decline drifts, production drifts, level developments, and attack ramps. For intersections, cable anchors, in addition to the primary support, are required as well as for sections where the cut is >9 m wide.

Poor, and very-poor, rock mass conditions are expected in the Kakula South steeply-dipping area and are in accordance with visual observations made from the available borehole data (Kakula South presentation by G. Gilchrist, 20220325). In these areas, cable anchors and mesh reinforced shotcrete will have to be installed, in addition to the recommended primary support.

16.1.5 Ground Support Recommendations

Ground support requirements have been determined, and the recommended support types are shown in Table 16.4 to Table 16.9.

Table 16.4 Drift-and-Fill Mining

	Support	
Good Conditions	Primary Drift	3.0 m long resin bolt in a 1.5 m x 1.5 m pattern on the hanging wall and the sidewalls. + 25 mm of fibre reinforced shotcrete or 10 mm V-seal.
	Secondary Drift	3.0 m long resin bolt in a 1.5 m x 1.5 m pattern on the hanging wall and sidewall. *3 x 6.5 m long cable anchors in accordance with agreed pattern. + 25 mm of fibre reinforced shotcrete or 10 mm V-seal.
	Tertiary Slipe	3.0 m long resin bolt in a 1.5 m x 1.5 m pattern on the hanging wall. + 25 mm of fibre reinforced shotcrete or 10 mm V-seal.
Fair Conditions	Primary Drift	3.0 m long resin bolt in a 1.5 m x 1.5 m pattern on the hanging wall and the sidewalls. + 25 mm of fibre reinforced shotcrete or 10 mm V-seal.
	Secondary Drift	3.0 m long resin bolt in a 1.5 m x 1.5 m pattern on the hanging wall and sidewall. *3 x 6.5 m long cable anchors in accordance with agreed pattern. + 25 mm of fibre reinforced shotcrete or 10 mm V-seal.
	Tertiary Slipe	3.0 m long resin bolt in a 1.5 m x 1.5 m pattern on the hanging wall. + 25 mm of fibre reinforced shotcrete or 10 mm V-seal.
Poor Conditions	Primary Drift	3.0 m long resin bolt in a 1.0 m x 1.0 m pattern on the hanging wall and the sidewalls. + 50 mm of mesh reinforced shotcrete.
	Secondary Drift	3.0 m long resin bolt in a 1.0 m x 1.0 m pattern on the hanging wall and sidewall. *3 x 6.5 m long cable anchors in accordance with agreed pattern. + 50 mm of fibre reinforced shotcrete.
	Tertiary Slipe	3.0 m long resin bolt in a 1.0 m x 1.0 m pattern on the hanging wall. + 50 mm of fibre reinforced shotcrete.
Very Poor Conditions	Primary Drift	3.0 m long resin bar bolt in a 1.0 m x 1.0 m pattern on the hanging wall and the sidewalls. + 75 mm of fibre reinforced shotcrete.
	Secondary Drift	3.0 m long resin bar bolt in a 1.0 m x 1.0 m pattern on the hanging wall and sidewall. *3 x 6.5 m long cable anchors in accordance with agreed pattern. + 75 mm of fibre reinforced shotcrete.
	Tertiary Slipe	3.0 m long resin bar bolt in a 1.0 m x 1.0 m pattern on the hanging wall. + 75 mm of fibre reinforced shotcrete.
Extremely Poor Conditions	Primary Drift	3.0 m long resin bar bolt in a 0.8 m x 0.8 m pattern on the hanging wall and the sidewalls. Mesh 100 mm x 100 mm x 5 mm or approved mesh available on-site. + 100 mm of fibre reinforced shotcrete.
	Secondary Drift	3.0 m long resin bar bolt in a 0.8 m x 0.8 m pattern on the hanging wall and sidewall. Mesh 100 mm x 100 mm x 5 mm or approved mesh available on-site. *3 x 6.5 m long cable anchors in accordance with agreed pattern. + 100 mm of fibre reinforced shotcrete.
	Tertiary Slipe	3.0 m long resin bar bolt in a 0.8 m x 0.8 m pattern on the hanging wall. Mesh 100 mm x 100 mm x 5 mm or approved mesh available on-site. + 100 mm of fibre reinforced shotcrete.

Figure 16.12 illustrates the recommended support configuration in accordance with Figure 16.11, which confirms that a 3 m x 3 m cable anchor spacing, with one anchor installed at a 60° angle (into the tertiary slipe), in conjunction with the primary support 1.5 m x 1.5 m rebar will provide sufficient support for the secondary and tertiary drives.

Figure 16.13 illustrates the section view.

Figure 16.12 Recommended Support Configuration for the Secondary and Tertiary Drives

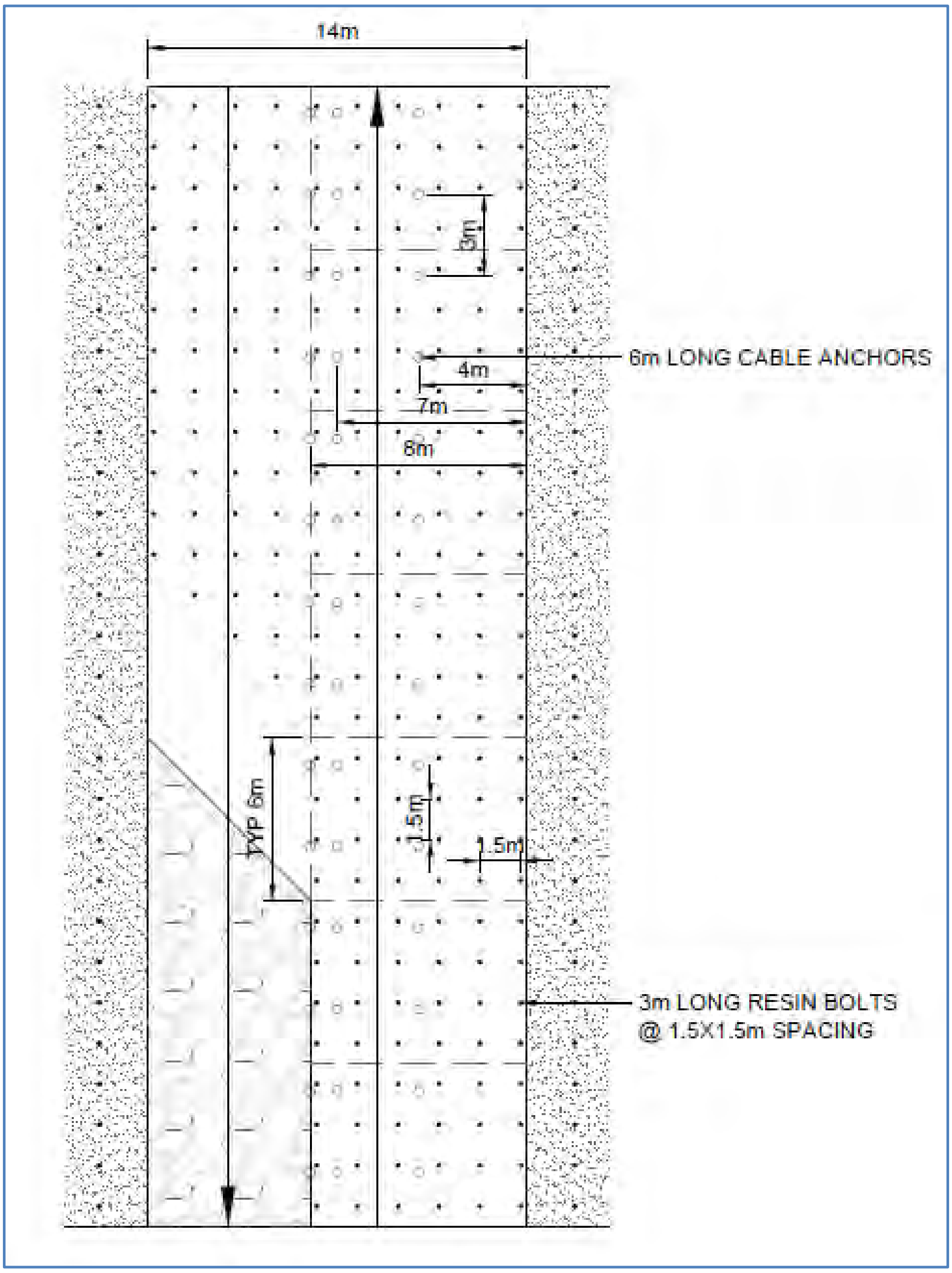


Figure 16.13 Recommended Support Configuration for the Secondary and Tertiary Drives – Section View

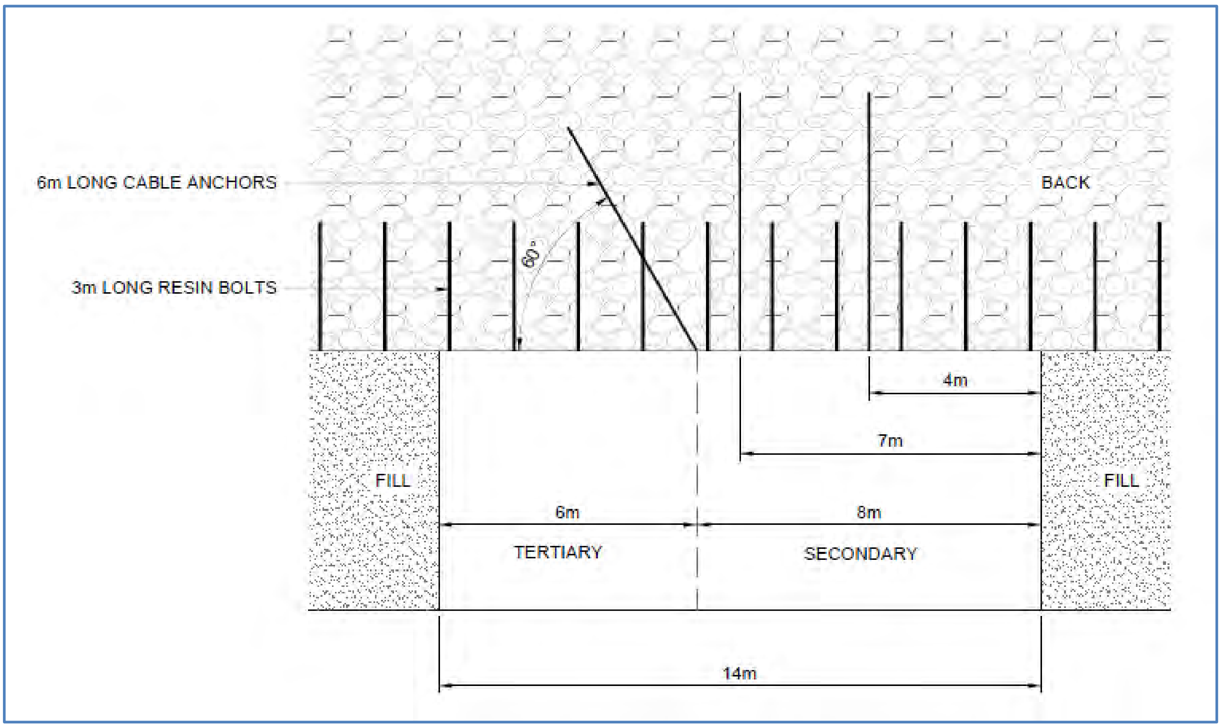


Table 16.5 Access, Perimeter Drifts, Conveyor Drive and Workshops

Conditions	Support
Perimeter Drifts (Diamictite)	
Good conditions	3.0 m long resin bar bolt or split set (depending on ground water) in a 1.5 m x 1.5 m pattern on the hanging wall and the sidewalls. + 25 mm of fibre reinforced shotcrete or 10 mm V-seal.
Fair conditions	3.0 m long resin bar bolt or split set (depending on ground water) in a 1.0 m x 1.5 m pattern on the hanging wall and the sidewalls. + 25 mm of fibre reinforced shotcrete or 10 mm V-seal.
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 fibre reinforced shotcrete.
Extremely poor conditions	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 mesh reinforced shotcrete.
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 fibre reinforced shotcrete or 10 mm V-seal.
Fair conditions	3.0 m long resin bar bolt in a 1.0 m x 1.5 m pattern on the hanging wall and the sidewalls. + 50 m of fibre reinforced shotcrete or 10 mm V-seal.
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 fibre reinforced shotcrete.
Extremely poor conditions	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 mesh reinforced shotcrete.
Intersections	6.5 m long cable anchors (pattern offset from primary support).
Underground Workshops (Diamictite)	
Good conditions	3.0 m long resin bar bolt or split set (depending on ground water) in a 1.5 m x 1.5 m pattern on the hanging wall and the sidewalls. + 25 mm of fibre reinforced shotcrete or 10 mm V-seal. Where exceeding 6 m excavation span, 6.5 m long cable anchors at 2.0 m x 2.0 m spacing to be installed.
Fair conditions	3.0 m long resin bar bolt or split set (depending on ground water) in a 1.0 m x 1.5 m pattern on the hanging wall and the sidewalls. + 25 mm of fibre reinforced shotcrete or 10 mm V-seal. Where exceeding 6 m excavation span, 6.5 m long cable anchors at 2.0 m x 2.0 m spacing to be installed.
Intersections	6.5 m long cable anchors (pattern offset from primary support) and for the excavations where width is ≥8.5 m.
Note that underground workshops should not be constructed in poor or extremely poor rock conditions. Fair ground can be considered but good conditions are preferred for underground workshops.	

Table 16.6 Twin Connection Drifts (depth <500 m)

Conditions	Support
Good conditions	3.0 m long resin bar bolt or split set (depending on ground water) in a 1.5 m x 1.5 m pattern on the hanging wall and the sidewalls. + 25 mm of fibre reinforced shotcrete or 10 mm V-seal.
Fair conditions	3.0 m long resin bar bolt or split set (depending on ground water) in a 1.0 m x 1.5 m pattern on the hanging wall and the sidewalls. + 25 mm of fibre reinforced shotcrete or 10 mm V-seal.
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 fibre reinforced shotcrete.
Extremely poor conditions	3.0 m long resin bar bolt in a 1.0 m x 1.0 m pattern on the hanging wall and the sidewalls. + 75 mm of mesh reinforced shotcrete.
Intersections	6.5 m long cable anchors (pattern offset from primary support).

Table 16.7 Twin Connection Drifts (depth >500 m)

Conditions	Support
Good conditions	3.0 m long resin bar bolt or split set (depending on ground water) in a 1.5 m x 1.5 m pattern on the hanging wall and the sidewalls. + 25 mm of fibre reinforced shotcrete or 10 mm V-seal.
Fair conditions	3.0 m long resin bar bolt or split set (depending on ground water) in a 1.0 m x 1.5 m pattern on the hanging wall and the sidewalls. + 25 mm of fibre reinforced shotcrete or 10 mm V-seal.
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 fibre reinforced shotcrete.
Extremely poor conditions	3.0 m long resin bar bolt in a 1.0 m x 1.0 m pattern on the hanging wall and the sidewalls. + 75 mm of mesh reinforced shotcrete.
Intersections	6.5 m long cable anchors (pattern offset from primary support).

Table 16.8 Hanging Wall Access Drift-and-Fill (steeply-dipping areas)

Conditions		Support
Good conditions	Twin Decline drifts or single decline	3.0 m long resin bar bolt in a 1.5 m × 1.5 m pattern on the hanging wall and the sidewalls. + 50 mm of fibre reinforced shotcrete.
	Level Developments	3.0 m long resin bar bolt in a 1.5 m × 1.5 m pattern on the hanging wall and the sidewalls. + 50 mm of mesh reinforced shotcrete.
	Attack ramps	3.0 m long resin bar bolt in a 1.5 x 1.5 m pattern on the hanging wall and the sidewalls.
	Intersections	Primary support + cable bolting (pattern 3,0 x 2,5 m).
	Production drifts	3.0 m long split set bolt in a 1.5 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	3.0 m long resin bar bolt in a 1.5 m × 1.0 m pattern on the hanging wall and the sidewalls. + 50 mm of fibre reinforced shotcrete.
	Level Developments	3.0 m long resin bar bolt in a 1.5 m × 1.0 m pattern on the hanging wall and the sidewalls. + 50 mm of mesh reinforced shotcrete.
	Attack ramps	3.0 m long resin 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 x 2,5 m).
	Production drifts	3.0 m long split set bolt in a 1.5 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 Conditions	Twin Decline drifts or single decline	3.0 m long resin bar bolt in a 1.0 m × 1.0 m pattern on the hanging wall and the sidewalls. + 50 mm of fibre reinforced shotcrete.
	Level Developments	3.0 m long resin bar bolt in a 1.0 m × 1.0 m pattern on the hanging wall and the sidewalls. + 50 mm of mesh reinforced shotcrete.
	Attack ramps	3.0 m long resin 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 x 2,0 m).
	Production drifts	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 (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 recognise ground conditions; 75 mm fibre reinforced shotcrete high strength wearing resistant mix (HSWR).

Table 16.9 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. Pre-tensioning required. 6.5 m long.
Splitset bolts	3CR12.
Mesh	Black welded steel mesh, minimum 5 mm gauge, maximum 100 mm aperture, when required. Galvanised mesh is recommended for corrosive environments (KPS).
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.
Shotcrete	Shotcrete, minimum 30 MPa strength (28 days) or V-Seal Rock liner (10 mm).
OSRO straps	Steel straps, 250 mm wide, configuration with 4 rods, rod thickness 8–10 mm.

16.1.6 Overall Mine Stability

Elastic numerical modelling was carried out to review the LOM mining plan during the Kakula FS study. The overall mine layout has not changed. The review was carried out to assess the stability of the following excavations:

- Perimeter drifts
- Connection drifts
- Double cut mining
- Subsidence

16.1.6.1 Input Parameters

The elastic properties used in the models were obtained from the laboratory test results and rock mass classification presented in the SRK memo "Kakula (PFS) Rock Strength Properties" (SRK, 2018a). The rock mass modulus for the overburden material was determined by downgrading the intact modulus using the methods published by Hoek and Diederichs (2006). Using backfill strength of 90 kPa obtained from free standing strength requirements (SRK Memo "Ground support requirements (SRK, 2018b)), the equivalent strain was estimated from the typical backfill-hyperbolic curve. Assuming a linear relationship the backfill modulus of 0.050 GPa was determined. A Poisson's ratio of 0.35 which is typical of free standing backfill, and this was assumed for the backfill. Table 16.10 summarises the elastic material properties.

The virgin state of stress was determined for the overburden, using a rock density of 2,770 kg/m^3, and k-ratio of 1.0, as presented in Table 16.11.

Table 16.10 Material Properties

Item	Rock Mass Modulus	**Poisson's Ratio**	Intact rock UCS for orebody (Combination of SDT and SST)
Hanging wall	25 GPa	0.27	85 MPa
Fill material	0.050 GPa	0.35	150 to 200 kPa

Table 16.11 Stress Gradient

S_{xx} (MPa/m)	S_{yy} (MPa/m)	S_{zz} (MPa/m)
0.02770	0.02770	0.02770

16.1.6.2 Model Geometry and Analysis

The LOM design layout was provided by Kamoa Copper SA in .DXF file format. The Boundary Element Method (BEM) of stress analysis assumes that the rock is elastic, homogeneous and continuous. Each model was constructed using Fictitious Force (FF) elements. FF elements are three-dimensional blocks used to simulate the drifts. The production areas were modelled using Fictitious Force (FF) elements to assess the stress influence on the drifts.

Note that for the assessment of the perimeter drifts the drift-and-fill stope were assumed to be backfilled.

The connection drifts will provide access to the production drifts, and for ventilation purposes. The connection drifts will be constructed within the protection pillars and will be required to be in serviceable condition until the end of life of the stopes. The protection pillars, and the connection drifts, will experience stress change as drift extraction progresses.

Assessment Criterion

The potential damage for the perimeter drifts was assessed using the Depth of Fracturing (DF) (Martin et al., 1999 and Cai and Kaiser, 2014):

$$\boldsymbol{DF = \frac{D_e}{2}\left(\frac{1.25(3\sigma_{1i} - \sigma_{3i})}{\sigma_c} - 0.51\right)}$$

Where:

- D_e= excavation width (6 m).
- σ_{1i} and σ_{3i} are the major and minor principal stress components acting normal to the excavation.
- σ_c is the uniaxial compressive strength of the intact material of the ore horizon combination of SDT SST (85 MPa) for excavations at orebody horizon.

In elastic modelling the major and the minor principal stresses can be used with the Hoek Brown rock strength parameters to estimate the amount of damage due to stress (Map3D User Manual, 2018) in the protection pillars. The Strength Factor (SF) is determined as follows:

$$\boldsymbol{Strength\ Factor\ A} = [\,\boldsymbol{\sigma_3} + \sqrt{(\boldsymbol{m\sigma_c\sigma_3} + \boldsymbol{s\sigma_c^2})}]/\boldsymbol{\sigma_1}$$

Where:

- σ_1 and σ_3 are the major and minor principal stresses.
- σ_c is the uniaxial compressive strength (UCS) of the orebody.
- m and c are the Hoek-Brown parameters.

Hoek-Brown properties for the orebody were obtained from Rock Strength Properties and are summarised in Table 16.12. The rock mass Hoek-Brown strength parameters, mb and s, were estimated by downgrading the intact rock properties using an approach presented in Hoek et al., (2002). The rock mass scaling equations for mb and s are as follows:

$$\boldsymbol{m_b = m_i exp\left(\frac{GSI - 100}{28 - 14D}\right)}$$

$$\boldsymbol{s = exp\left(\frac{GSI - 100}{9 - 3D}\right)}$$

Where:

- GSI is the Geological Strength Index.
- D is the disturbance factor.

For the assessment of the protection pillars, the disturbance factor which is a measure of the blast damage and stress relaxation of the rock mass was assumed to be 0.1 (good blasting). The UCS values were determined from SRK laboratory test results with GSI values calculated from logging parameters according to the following formula.

$$\boldsymbol{GSI = \frac{52\frac{Jr}{Ja}}{1 + \frac{Jr}{Ja}} + \frac{RQD}{2}}$$

Table 16.12 Hoek-Brown Strength Parameters

Parameter	Unit
UCS	85 MPa - Estimated from SRK lab test results. (mean for SDT and SSL)
m_i	10
GSI	59
m_b	3.5
s	0.0205

16.1.6.3 Findings

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.

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

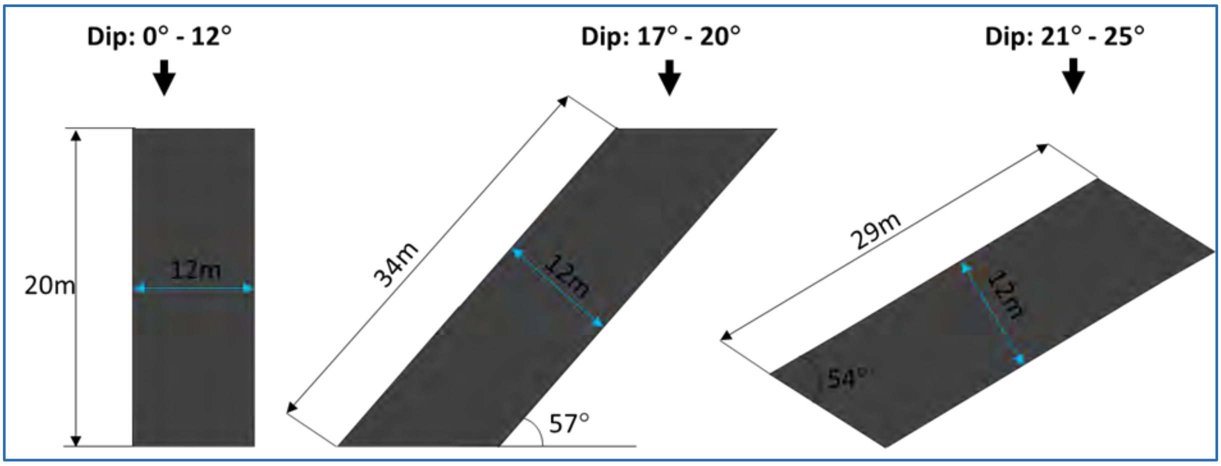
From a 500 m depth, the tertiary drifts-and-pillar are close to failure prior to the extraction of this drift. It must also be noted that the ore pillar prior to tertiary extraction will have a potential to fail throughout statically due to stress fracturing or dynamically by strain bursting. This situation 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 16.13 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 16.14 is an example for three of the dip assumptions 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 16.13 Connection Drifts and 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

Figure 16.14 Example of Production Pillar Configuration and Varying Orebody Dip



Production Pillar Extraction

The production pillar extraction strategy uses a retreat 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.

Double-Cut Mining

The second cut is scheduled to be selectively mined many years after the mining first cut. Providing that placement of backfill is well controlled during the cut, it is expected to be well consolidated. This will prevent deterioration of ground conditions due to relaxation and allow additional strength gain in the backfill.

If tight filling is not accomplished on the first lift horizon, significant bed separation, and ravelling, will be encountered when mining the second lift. Unfilled drifts will not only enhance bed separation, but there may also be a risk of personnel, and mobile machinery, falling into open voids. Rockfalls and overbreak will influence the second cut, particularly if the voids created cannot be properly filled. It is important that these incidents are marked on mine plans to enable the potential hazards to be identified prior to 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.

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.

Subsidence

Due to the contiguous placement of backfill, 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.

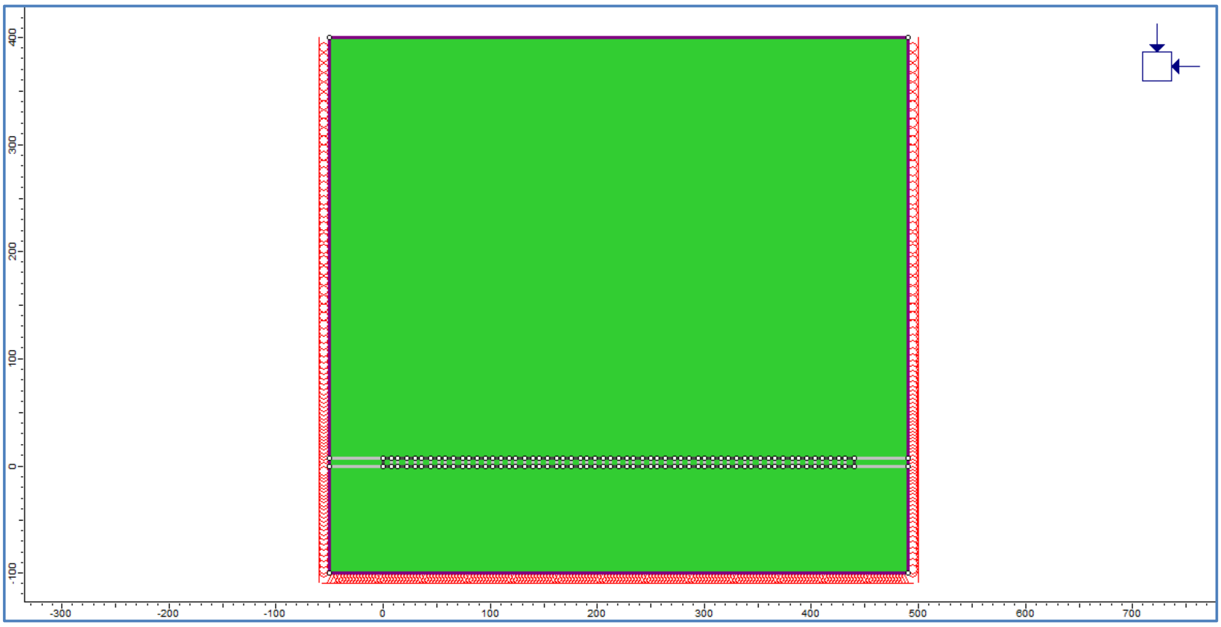
16.1.7 Mining Sequence and Backfill

The drift-and-fill mining sequences were modelled using Rocscience RS2, which is a two-dimensional (2D) finite element modelling program.

16.1.7.1 Model Layout and Sequence

The model layout is represented in Figure 16.15. For the purposes of this analysis, a depth of mining of 400 m was simulated. At greater depth, the closure will be greater, which may require higher backfill strengths. A total model mining span of 440 m was simulated. Since the model is 2D, the out of plane distance is infinitely long. Within the 7.6 m high mining cut, 8 m wide primary production drifts, 8 m wide secondary production drifts, and 6 m wide sliping production drifts are represented.

Figure 16.15 RS2 Model

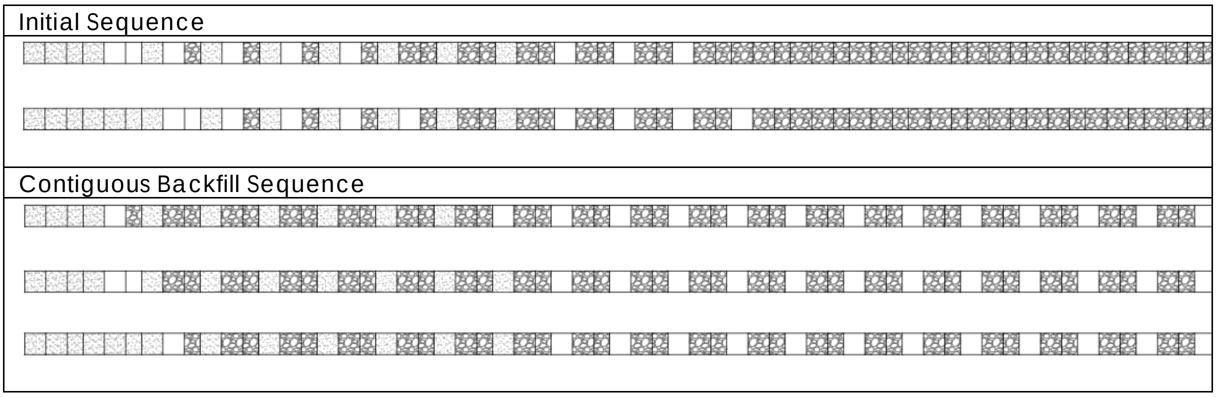


Two model mining sequences were simulated and are presented in Figure 16.16. The dark grey shading represents rock (Diamictite/Siltstone), light grey is backfill, and white, is mined out.

An initial sequence was provided for review, which incorporates four open secondary drifts to enable cable bolting. Two steps are presented in Figure 16.16, and this sequence is repeated with the drift sequence moving to the right. During each mining step, backfill is placed in one primary drift, and one primary secondary combination. This enables the mining of one primary, one secondary, and one tertiary, during each mining step. The sequence migrates to the right. During this sequence there will always be one 8 m wide unconfined backfill “pillar” that is formed. The completed mined area will always have contiguous fill in this sequence.

A more conservative sequence was also modelled (Contiguous backfill sequence). Three steps are shown in Figure 16.22, Figure 16.23, and Figure 16.24 to illustrate the sequence. Primary drifts need to be filled in advance to enable curing. Only one secondary drift is developed at a time, but cable bolt installation can lag behind the face, but must be installed prior to sliping. Tertiary sliping can take place as soon as the cable bolts are installed. The secondary and tertiary are backfilled simultaneously, which enables the next secondary drift to be developed. It is not necessary to wait for curing, but the empty void must be filled to provide the necessary confinement. This sequence ensures that no unconfined backfill pillars are ever formed and the backfill is always contiguous. Also, there are no open secondary drifts, which means that adjacent to the completed mined area there will always be 14 m wide solid rock pillars (unmined secondary and tertiary). This sequence is slower since it is not possible to develop a secondary drift and tertiary sliping simultaneously. This may be affected further by the cable bolting cycle. In this sequence, production will need to be made up by mining different areas simultaneously. The primary drifts can be mined in advance, but this will always result in a sequence where the secondary and tertiary mining is lagging.

Figure 16.16 Mining Sequences in the Model (Vertical Cross-Section)



16.1.7.2 Material Properties

In the model, only one rock type (Diamictite/Siltstone) is presented to simplify the model. At Kakula, the footwall is Roan sandstone, but this does not significantly affect pillar stability, nor hanging wall closure.

The intact rock material properties (Table 16.14) are from the Kakula FS study (Table 16.14).

Table 16.14 Intact Rock Properties (Diamictite/Siltstone)

Elastic modulus (GPa)	**Poisson's ratio**	UCS (MPa)	m_i	Density (kg/m^3)
60	0.26	95	13	2700

The rock mass properties were determined for fair, and good quality rock masses (Table 16.15). Elastic, perfectly plastic material behaviour (residual strength = peak strength) was applied in the model.

Table 16.15 Rock mass properties (Diamictite/Sandstone)

Rock quality	GSI		Modulus (GPa)	m_b	s	a
Fair	50		18.4	2.18	0.004	0.51
Good	65		37.9	3.73	0.020	0.50

Backfill properties (Table 16.16) were selected from the Patterson and Cooke, and OHMS backfill reports. Backfill was modelled as elastic, and the cohesion and friction were simply included to check the strength factors (Mohr-Coulomb). Lateral confinement due to contiguous backfill increases the strength of the backfill in the mined-out area.

In practice, it is expected that the backfill will exhibit some degree of strain hardening behaviour when it is well confined, due to consolidation. The stiffness of the backfill may therefore be greater than the initial 20 MPa modulus after significant closure has taken place. After a span of 11 x 7.6 m mining height = 84 m^2 has been mined and backfilled, it will not fail, based on experience in South Africa. Therefore, applying a constant modulus of 20 MPa is appropriate for unconfined backfill, but slightly conservative when the backfill is well confined. At this stage, it is not possible to determine the strain hardening characteristics until in situ measurements of closure and backfill load have been carried out.

Table 16.16 Backfill Properties

Modulus (MPa)	Cohesion (kPa)	Friction angle	UCS (kPa)
20	60	30	208

16.1.8 Model Results

Four models were analysed to determine the rock mass behaviour associated with the two sequences for fair, and good quality rock masses. The models stepped through the sequence, gradually increasing the completely mined and backfilled span until they became unstable and did not converge to a solution. At this point, it represents mass failure of overlying hanging wall.

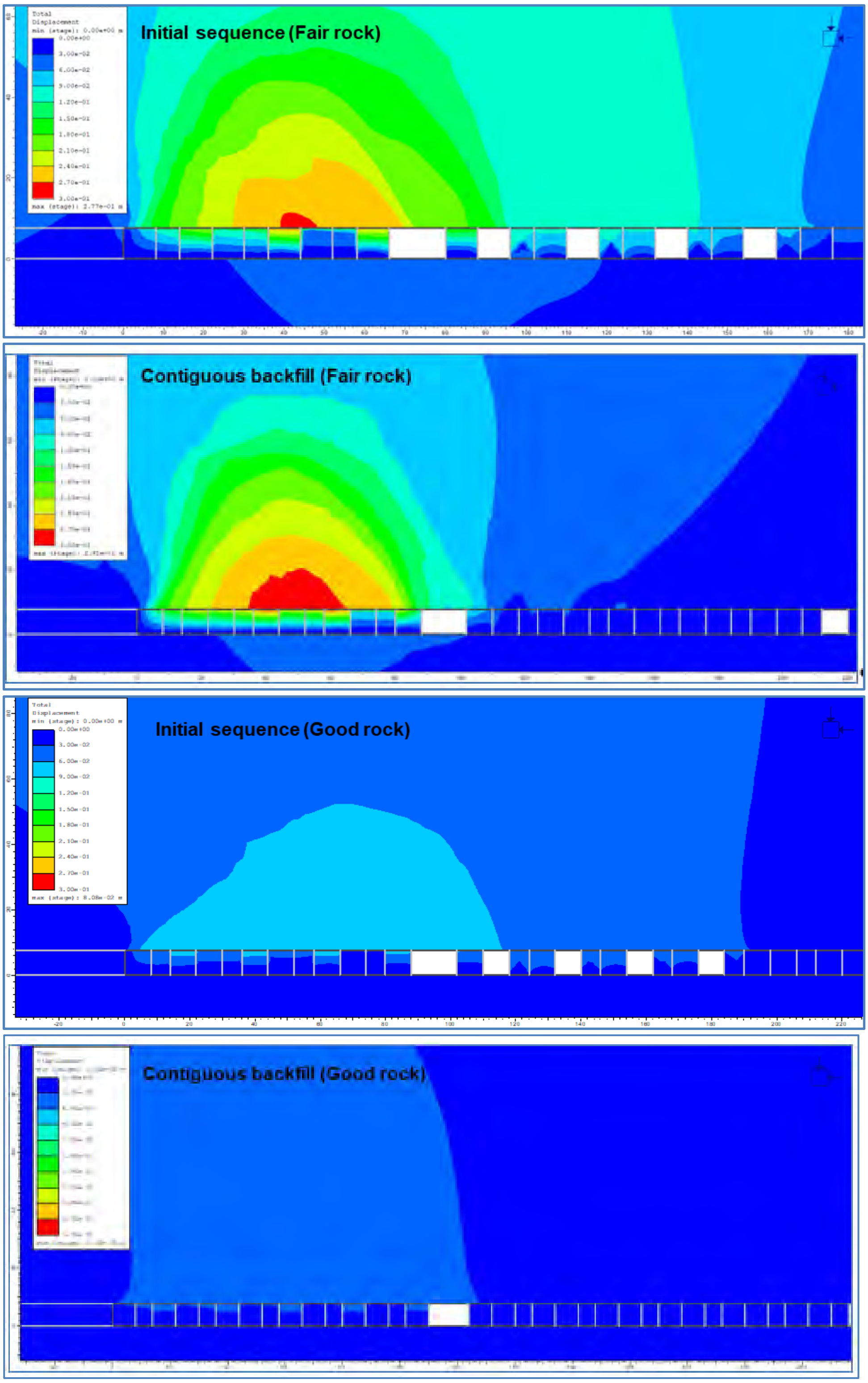
Figure 16.17 represents total displacement in the model at the mining step prior to mass failure. The displacements represent closure or downward movement (elastic and plastic) of the overlying rock mass. In reality, the plastic displacement would be due to bedding separation, separation and sliding along joints, and tensile failure of intact rock. Prior to the model becoming unstable, the plastic strain is less than 2%, which indicates that the rock mass can be supported.

As expected, the fair quality rock mass exhibits significantly more displacement than the good quality rock mass. The maximum displacement in the centre of the mined-out area exceeds 0.27 m for fair quality rock and is less than 0.09 m for good quality rock. Note that in practice, this may be mitigated by strain hardening in the backfill. The initial sequence also shows significant displacement above the mined secondary drifts. This is due to yielding of the tertiary pillars, which still need to be sliped. Horizontal deformation of up to 100 mm can be expected, indicating the requirement for yielding support.

The critical areas are the open secondary, and tertiary drifts. Backfill in the walls of open drifts will be unconfined, and backfill spalling can be expected, when the displacement causes the load to exceed the UCS. In the initial sequence, the backfilled primary stopes become 8 m wide backfill pillars (width to height ratio of about 1), which are likely to fail completely if the load exceeds the backfill UCS.

Note that the deformation occurs over a large volume, and the rebar deformation over the 3 m length is less than 30 mm and cable deformation over the 6 m length is less than 50 mm in the worst case. End anchored cables will accommodate the deformation.

Figure 16.17 Modelled Displacement at Maximum Span for each Scenario



The completely mined (primary, secondary, and sliping), and backfilled span, is tracked as the model progresses. Closure (displacement of the back) increases as the span is increased. Since this is a 2D model, and in reality, there will be ramp pillars, or other pillars that limit the out of plane span or length, the equivalent hydraulic radius (HR) is determined to make the findings more versatile:

HR = Area /Perimeter

The HR concept is used with empirical methods to represent mining dimensions, and to assess the stability of stopes, or caveability, in block cave mines. For an infinitely long excavation, the HR = Span/2. The chart (Figure 16.18) can be used to estimate the HR for mining block lengths and spans between solid rock pillars.

Figure 16.18 Maximum spans between Rock Pillars based on Hydraulic Radius

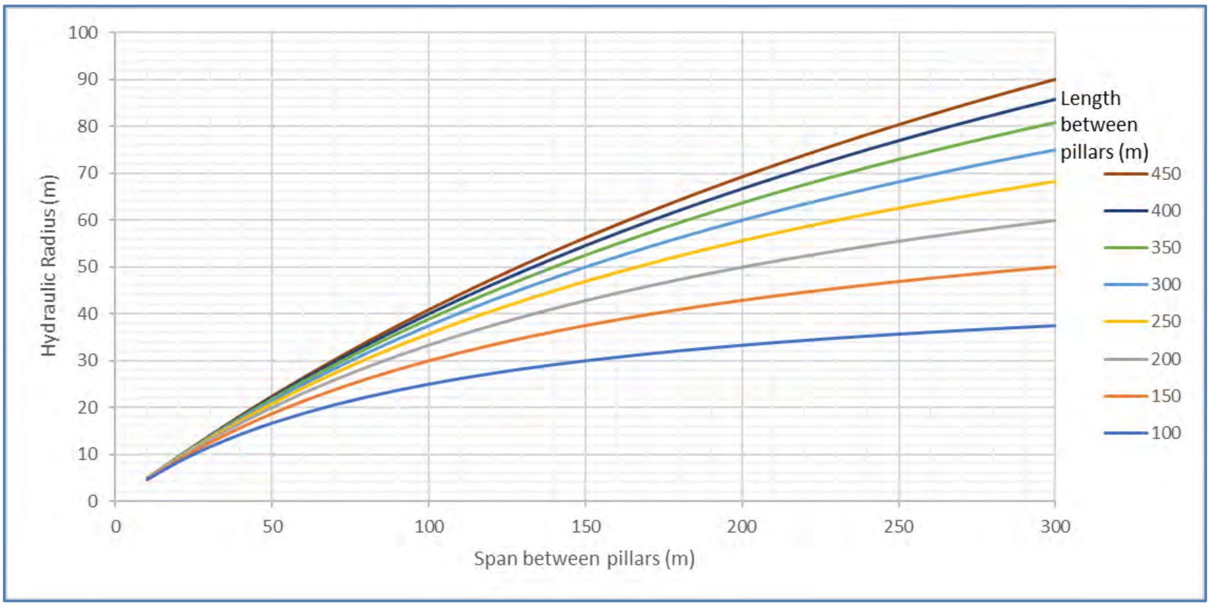


Figure 16.19 shows the displacement recorded during each step at the critical point where the backfill will be loaded. The backfill load is calculated as follows:

Backfill load = E δ/h

Where:

- E is the backfill elastic modulus (20 GPa).
- δ is recorded displacement.
- h is the mining height (7.6 m).

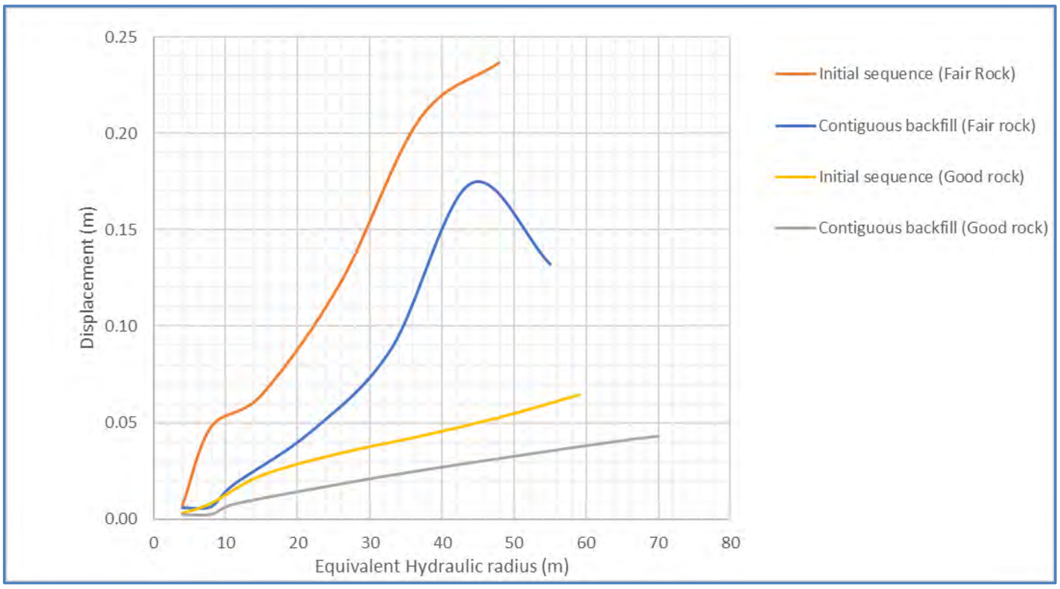
The backfill load increases as the HR of the completely mined area increases. The rate of increase of a fair rock mass is much greater, since it has a lower elastic modulus, and more plastic deformation occurs. When the backfill load exceeds the UCS, it will result in spalling of the backfill sides. In the contiguous backfill sequence, the higher deformation and load on the left wall is always indicated. The right-hand side will always be lower. For the initial sequence, failure of the unconfined backfill pillar may occur if load exceeds the UCS. The greater deformation and backfill loads in the initial sequence is due to the secondary drifts that are developed in advance, and yielding of the tertiary pillars, which will be sliped in the future. The analysis indicates that for fair rock conditions, a backfill UCS of 500 MPa and 650 MPa will be required for the contiguous backfill sequence, and initial sequences respectively.

Table 16.17 indicates the maximum spans (one step before failure of overlying rock) and equivalent HR for the different cases. The maximum spans and drift lengths need to be bounded by solid pillars, such as:

- Access drive pillars.
- Unmined secondary and tertiary (14 m) with backfill on either side.
- Specially designed pillars.

The estimated extraction ratio versus HR is plotted in Figure 16.20.

Figure 16.19 Displacement and Backfill Load versus Hydraulic Radius



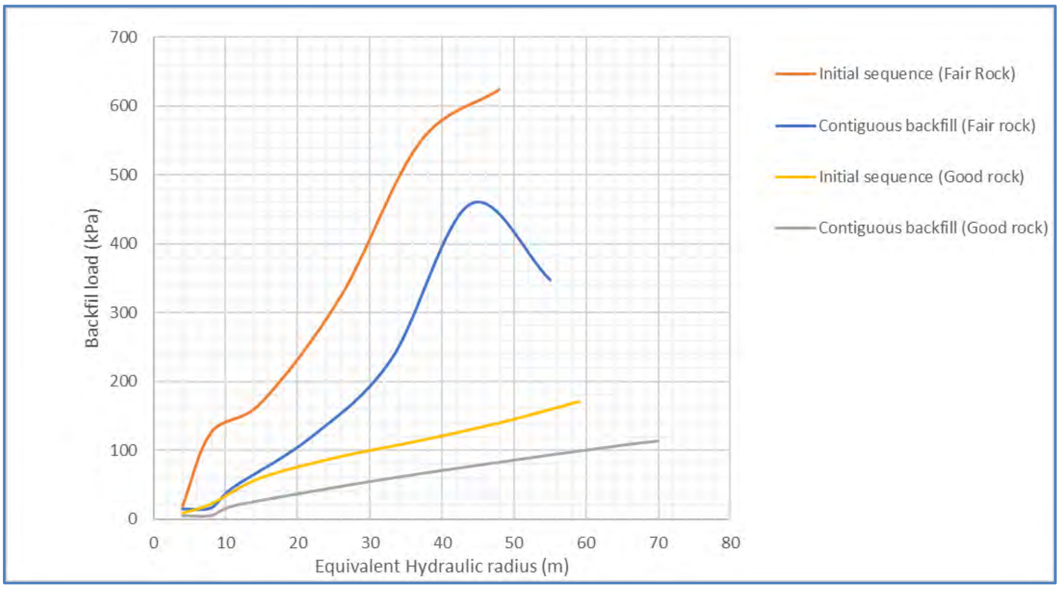
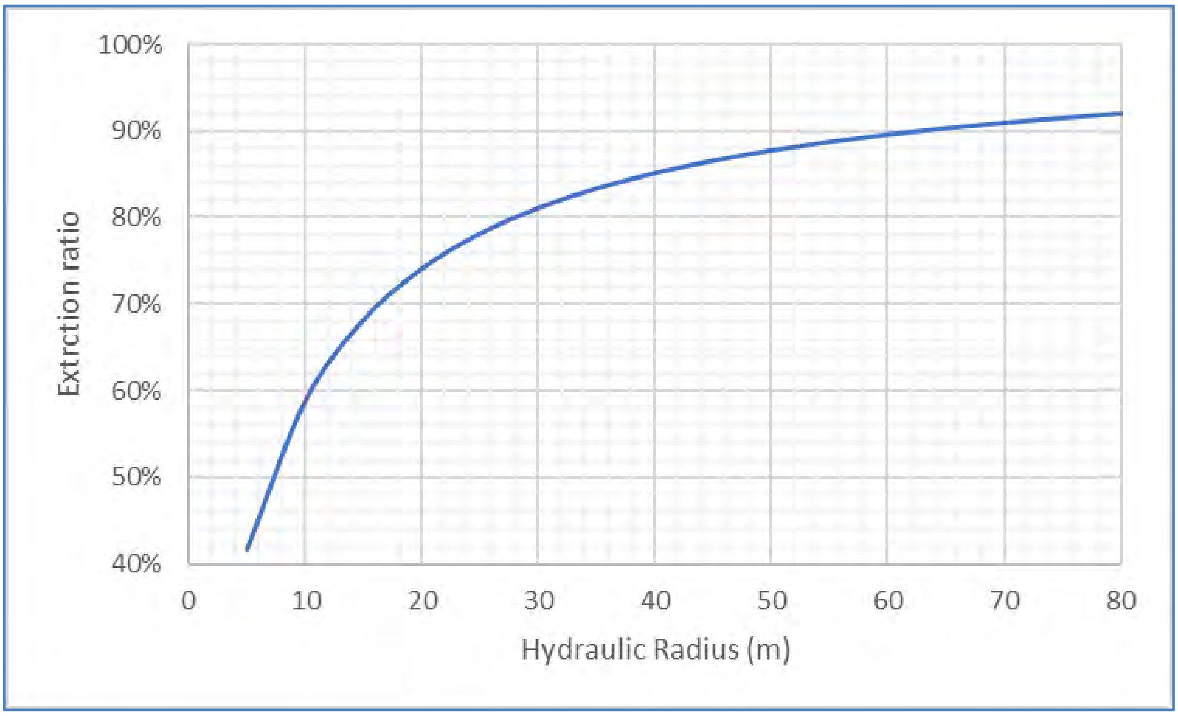


Table 16.17 Maximum Spans Before Hanging Wall Failure

Sequence	Rock Quality	Maximum Span (m)	Equivalent Hydraulic Radius (m)
Initial Sequence	Fair	96	48
Contiguous backfill	Fair	110	55
Initial Sequence	Good	118	59
Contiguous backfill	Good	132	66

Figure 16.20 Extraction Ratio versus Hydraulic Radius



16.1.9 Backfill Requirements

It is important to note that the backfill in the primary drifts is critical in terms of both strength, and effective tight filling. During the sequence, the loading on the backfill in primary drifts will increase as shown in Figure 16.19, and personnel will be exposed to potential spalling of the backfill during secondary drifting, and tertiary sliping. The backfill loading will depend on the rock mass quality.

It is recommended that the backfill in the primary drifts must have a strength of 500 kPa, and greater than 90% tight filling by area must be achieved. This strength requirement may be revised, when in situ testing shows that the strengths can be consistently achieved, backfill exposures are performing well in practice and underground closure measurements have been carried out in primary drifts.

Backfilling of the secondary drift, and tertiary sliping, is essential to ensure confinement of the backfill and pillars, to ensure stability of the active drifts, and crosscuts. The backfill strength is less critical since there will never be backfill exposures. To ensure that excess water is hydrated, it is recommended that a minimum backfill strength of 150 kPa is consistently achieved. While tight filling remains important, the tolerance can be reduced to 70% of the area. This will also minimise exposure of personnel in the 14 m wide span.

16.1.10 Proposed Layout and Sequence

Figure 16.21 shows crosscut pillars that should be left at the ends of the tertiary drifts. These will assist with the construction of backfill barricades and will control the stability of the crosscuts. They will be confined by backfill on three sides and will require yielding support in the wall adjacent to the crosscut. However, it important to note that because these pillars will yield, they do not reduce the effective HR (see Figure 16.18 and Table 16.17).

The recommended mining sequence for the drift-and-fill method is the contiguous backfill sequence, which is illustrated in Figure 16.22 to Figure 16.24. Figure 16.24 shows that the primary drifts need to be filled well in advance of the secondary drifts, to enable curing. A secondary drift is developed. Figure 16.23 shows the sliped tertiary. The secondary and tertiary must then be completely filled. The next secondary can then be mined (Figure 16.24), and the sequence is repeated. Only one secondary and tertiary combination may be open at any time. Backfilling must take place prior to development of the next combination.

It will therefore be necessary to move the mining crew to another block to maintain production, while the secondary tertiary combination is being filled.

Regional pillars will be required to prevent instability of the hanging wall.

Table 16.17 and Figure 16.18 should be used to determine the maximum spans between regional pillars. Regional pillars can be an unmined secondary and tertiary, which is confined by the backfilled primary drifts, pillars on access drives or any other planned pillar, which is more than 14 m wide and confined by backfill. The resulting extraction ratio can be estimated from Figure 16.20.

During modelling of the mining sequence only one direction for sliping was considered. This will however not have an effect on the outcome of the results since the direction of sliping can be adjusted once actual mining commences. Regardless of the sliping direction, there will always be an unconfined backfill ‘pillar’.

Figure 16.21 Crosscut Pillars

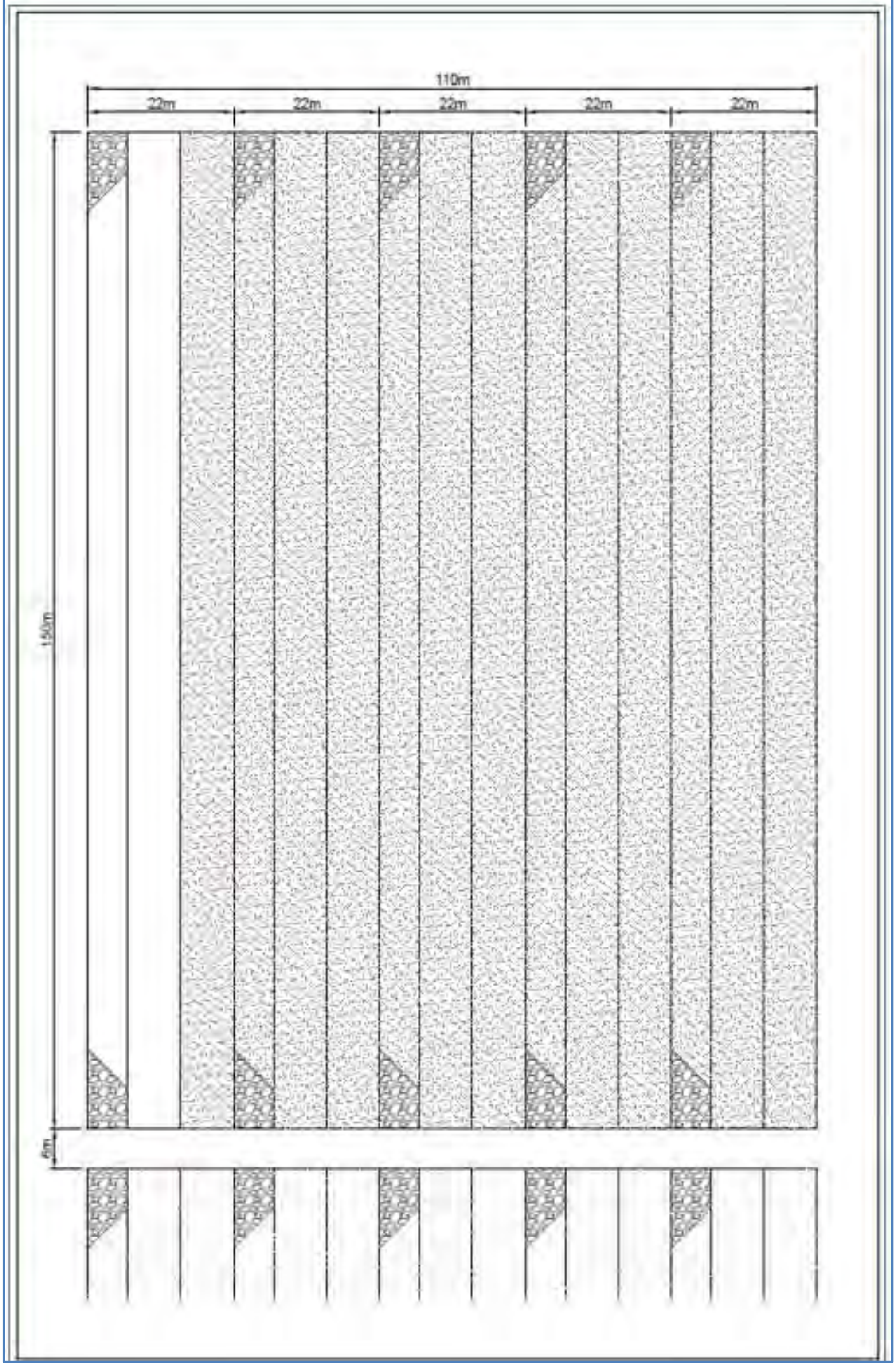


Figure 16.22 Contiguous Backfill Mining Sequence – Step 1

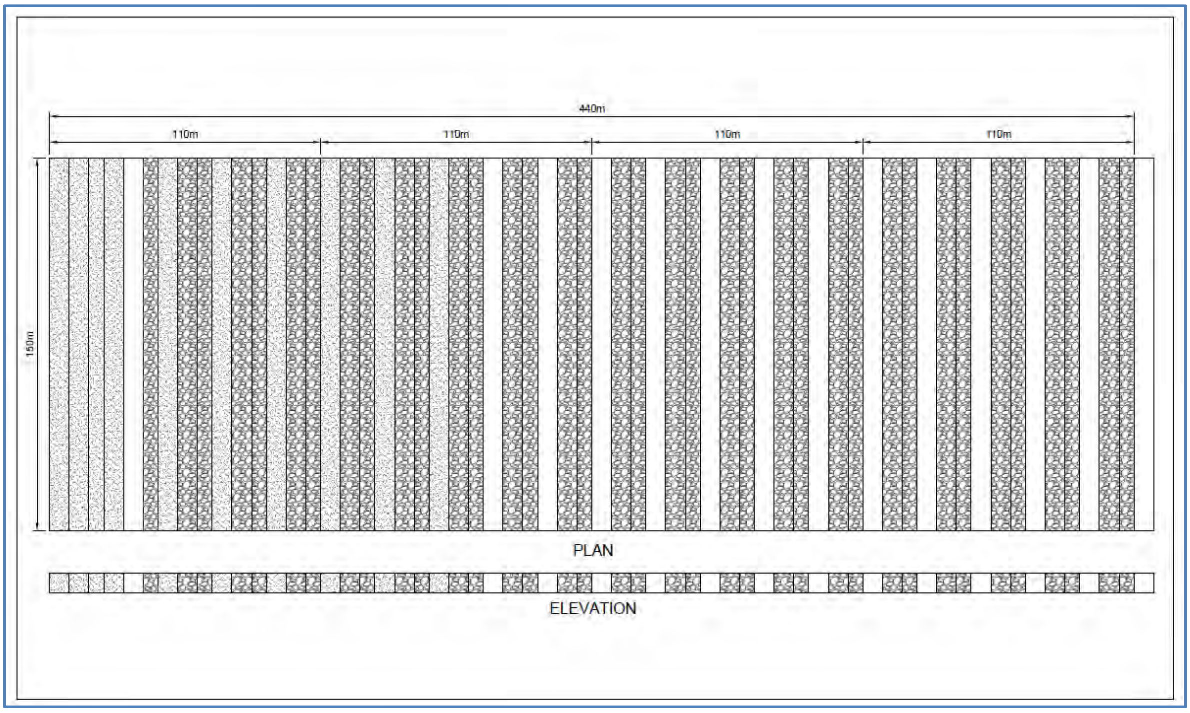


Figure 16.23 Contiguous Backfill Mining Sequence – Step 2

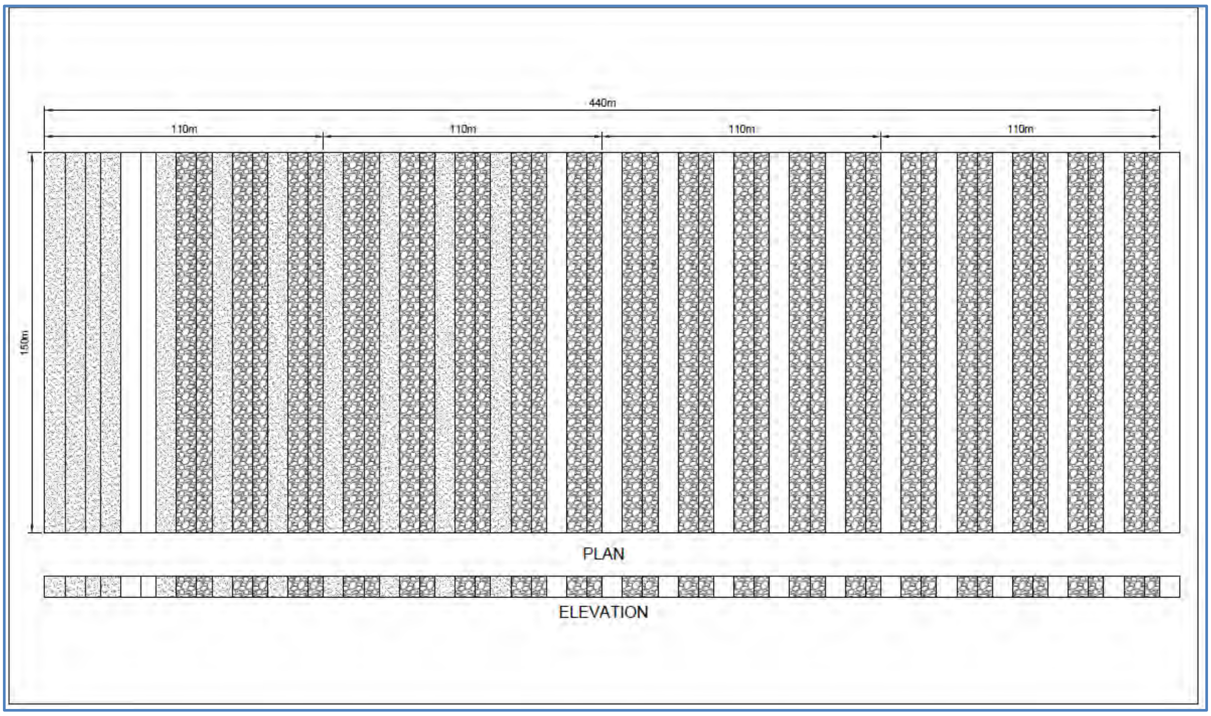
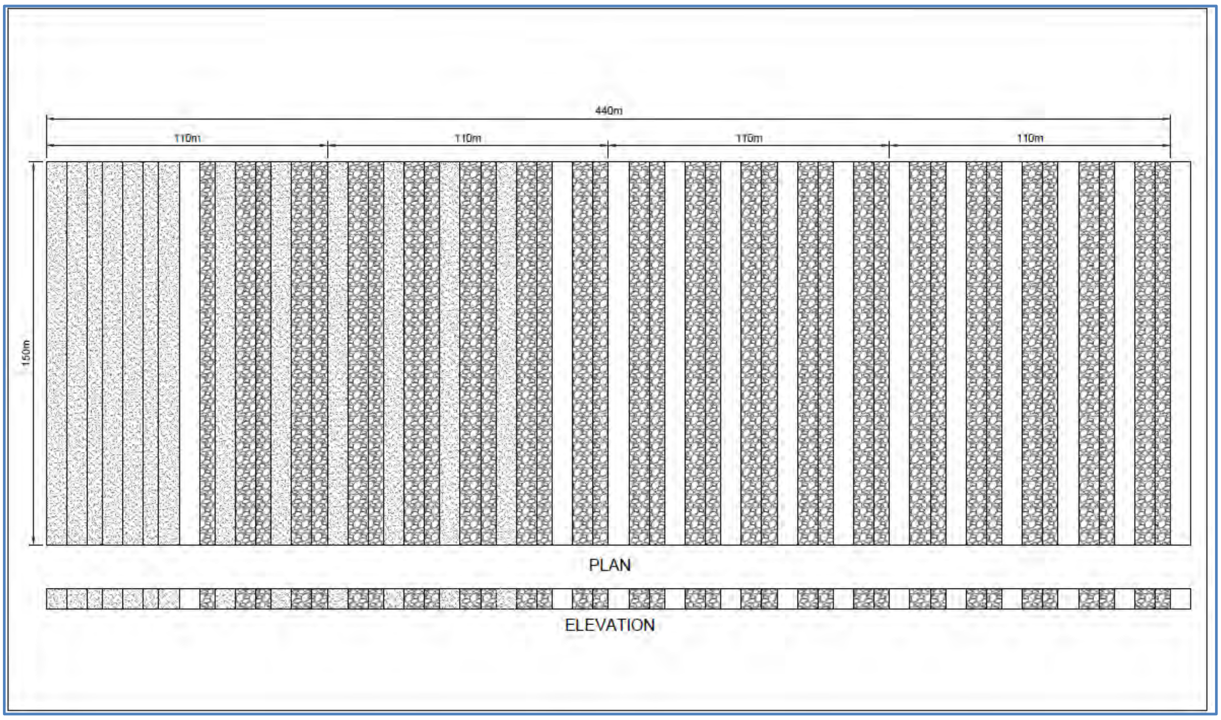


Figure 16.24 Contiguous Backfill Mining Sequence – Step 3



Drift-and-fill with sliping in poor to extremely-poor rock conditions can affect safe mining practice and conventional drift-and-fill without sliping should then be considered.

16.1.11 Further Geotechnical Considerations

16.1.11.1 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.

It is recommended that the backfill in the single cut primary drifts must have a strength of 500 kPa, and greater than 90% tight filling by area must be achieved. This strength requirement may be revised, when in situ testing shows that the strengths can be consistently achieved, backfill exposures are performing well in practice and underground closure measurements have been carried out in primary drifts.

For upper cut, and bottom cut where the orebody thickness is >6 m, and double lift mining is required, 200 kPa UCS will be sufficient.

Up dip or down dip extraction, and backfilling scenarios, as well as sequential backfilling in shorter sections should be considered to 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.

16.1.11.2 Monitoring of Potential Seismic Activity

The major structures, namely the west fault structures that 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.

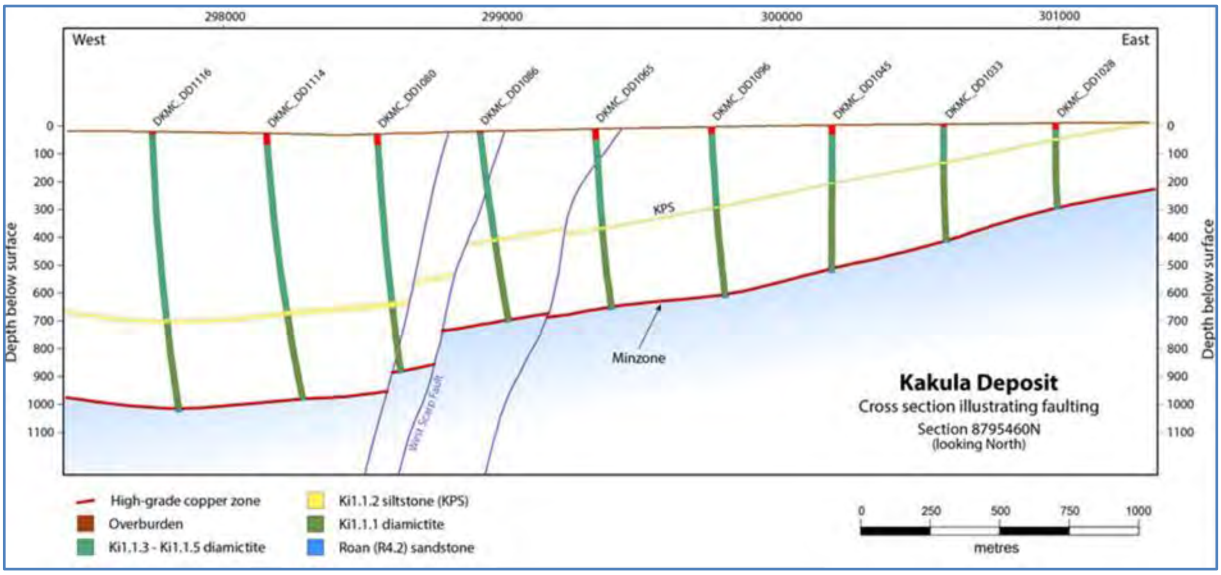
16.1.11.3 Analysis of Stand-Off Protection Zone for the West Scarp Fault

Major structures observed at Kakula share many characteristics with the structures observed at the Kamoa project, most notably the steep dip of the faults, the dominant north–north-east trending broad anticline, and the alignment of topographic features with brittle structures (Kakula 2017 Resource update).

In accordance with the Kakula 2017 resource update report, the most prominent faults at Kakula are three north–north east trending structures including, and related to, the West Scarp Fault (Figure 16.25) The West Scarp Fault, and a second fault approximately 150 m to the west, are steeply-dipping (approximately 75° to the west) normal faults, and jointly account for an offset of 150 m to 200 m. Approximately 350 m to 400 m east of the West Scarp Fault, a third fault has been modelled which dips approximately 75° to the west, with a reverse offset of approximately 10 m.

These structures have been characterised in drill core by steep breccias, calcite veining, and broken core. No true thickness intersections of these fault zones have yet been attained in drilling. Additional drilling and modelling are planned by Ivanhoe geologists to further characterise these faults.

Figure 16.25 Cross-section illustrating Major Structures and their offsets in the Kakula Area



The following are the recommended dimensions of the bracket pillar around each fault:

- WS Main fault should have a 40 m bracket pillar.
- WS2 fault should have a 20 m bracket pillar.
- WS3 should have a 20 m bracket pillar.

Considering the overall soft infilling of these structures the potential for seismic activity on these faults is considered unlikely to occur during the mining operation. In addition, for shallow mining operations, a 10 m bracket pillar on either side of the structures, and the use of backfill, has been found on numerous occasions to be a suitable methodology to contain any rock bursting associated with seismic activity along structures.

16.1.11.4 Underground Development in Roan Sandstone

Any development planned in the Roan sandstone should be carefully considered since the sandstone is confirmed to be water bearing and will influence short-term construction and long-term operations.

16.1.11.5 Kamoa Pyritic Sulphide (KPS) Support

It is understood that mining at Kamoa 1, and Kamoa 2, will take place close to, or within, the Kamoa Pyritic Sulphide (KPS) strata which has proven to be acidic (corrosive) and raising the concern of early deterioration of support. Discussions are currently underway with suppliers to see what corrosion resistant supports are on the market that can be used in this acidic environment.

It is also expected that the KPS is highly weathered / brecciated, and a rigid support regime (for poor rock conditions at least) will have to be implemented.

16.1.12 Concluding Remarks

The current geotechnical assessment is based on existing geotechnical data from:

- 2011–2013 geotechnical investigation (by SRK Vancouver).
- 2020 geotechnical investigation (SRK SA).
- Underground face mapping data collected during 2020–2022 at Kakula, and Kansoko mines.

Limited geotechnical data is available for Kansoko, Kamoa 1, and Kamoa 2, and it is recommended that further geotechnical information is gathered in these areas by allowing for a formal geotechnical drilling and geotechnical logging programme. This will enhance the quality of the geotechnical model and provide more detail towards expected rock mass conditions and support requirements. The limited geotechnical data affects the way in which the data is interpolated during Q-contouring. This is evident from areas where it is confirmed (from photographs) that poor and worse rock conditions will be present but where it is not presented in the Q-contours since there is no related geotechnical data.

Further geotechnical investigation is also required for geologically complex areas (steeply-dipping Kakula South, and structurally altered Kakula West) where detailed structural analysis is required.

The support strategies were compiled based on the results of the Q-contour maps for each mining footprint. Only minor changes have been made to the support recommendations from the 2020 Technical Report. SRK has added a support category for extremely poor rock mass conditions, which have been identified from the Q-contour maps.

There was also the proposal from the client to consider the v-seal instead of fibre reinforced shotcrete. In principle, SRK does not object using the 10 mm v-seal in good and fair ground. It is however recommended that the fibre reinforced shotcrete and mesh is used in poor to extremely poor rock mass conditions until the performance of the v-seal in these rock mass conditions is investigated and confirmed.

The geotechnical evaluation of the drift-and-fill with sliping is addressed in detail in Section 16.1.7 of this report. Although the method is accepted in principle there is a concern that sliping in poor rock and worse conditions will not be possible. In this case, it will be necessary to convert to conventional drift-and-fill without sliping.

The current backfill specifications are recommended based on laboratory test results. At Kakula mine, backfill performance monitoring and in situ testing is work in progress, and based on the outcome of this further optimisation of the backfill strengths may be considered. Backfill performance monitoring will also be required in future for the other deposits since it is not currently a requirement at PFS level.

16.1.13 Potential Geotechnical Risks

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

- Insufficient geotechnical data specific to Kansoko; Kamoa 1, and Kamoa 2, giving low confidence in the spatial variability in rock mass quality. Poor, and very-poor areas may result in reduced production rates and increased OPEX due to ground increased support. Infill drilling is required to understand the risk.
- Insufficient geotechnical data specific to the West Scarp fault and Kakula West, which are structurally complex. There may be reduced production rates and increased costs due to increased support. Infill drilling is required to understand the risk.
- Possible instability during raise boring. Geotechnical drillholes are required for each raisebore shaft to manage the risk.
- Porosity and water bearing properties of the Roan sandstone which will affect mining. The deeply weathered and leached nature of the sandstone will require additional support during mining.
- In steeply-dipping areas, and highly weathered / fractured ground conditions, drift-and-fill with sliping might not be possible. Converting back to conventional drift-and-fill without sliping will then be required.
- The acidity of the groundwater in areas where the KPS is present will lead to the acceleration of deterioration of support and adequate corrosion resistant support will have to be used.
- Global supply chain disruptions which can lead to shortages of geotechnical support material.
- 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 to confirm the in situ stress state. The stress tests should be prioritised.
- 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 drift-and-fill with sliping mining operations, negatively impacting on the pillar and span stability. Failure to achieve good quality blasting will significantly affect pillar performance.

16.1.14 Site Visits

For the Kamoa-Kakula 2023 PFS, three site visits were arranged by KCSA, where Kakula and Kansoko underground operations were visited in conjunction with various meetings related to mining methods (including geotechnical support) and mining operations, and infrastructure in general.

The first site visit was conducted during the week of 3–11 March 2022, and attended by Mr. William Joughin, and Ms Elani Vorster. The second site visit was scheduled for the week of 10–17 June 2022, and was physically attended by Ms. Elani Vorster, with Mr. William Joughin joining the discussions online. The third site visit was concluded by Ms. Elani Vorster and Mr. William Joughin during the week of 31 October – 4 November 2022.

16.2 Mining Method

The primary mining method for the Kamoa-Kakula deposits (drift-and-fill) was selected based on the results of a trade-off study which compared six different variations of drift-and-fill mining to identify the most suitable. The drift-and-fill method will be utilised as the primary mining method for all Kamoa-Kakula deposits. Identified mining areas with a dip greater than 25° will be mined using the hanging wall access drift-and-fill (HWAD&F) method.

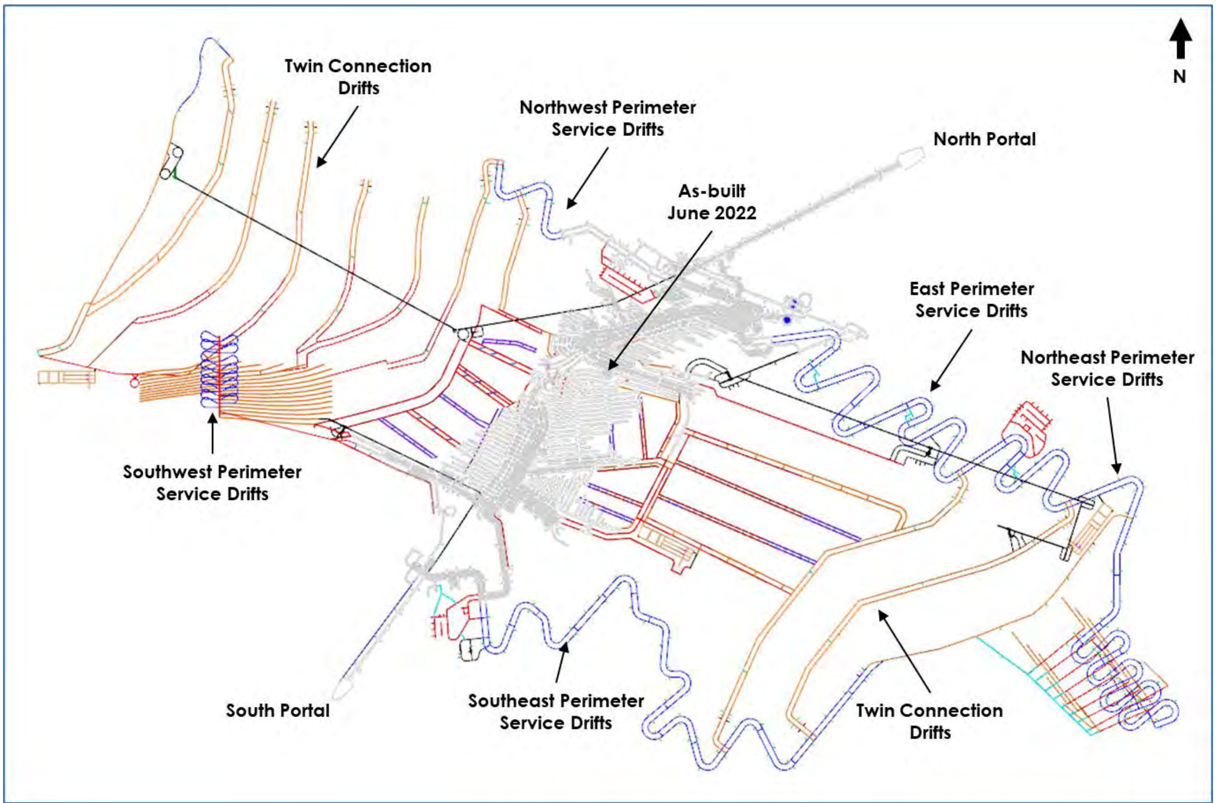
16.2.1 Mine Development Framework

Drift-and-fill mining is the primary method of extraction for the Kamoa-Kakula deposits. To establish the mining method, a pair of twin perimeter declines are driven at a defined offset to the extremities of each deposit. Twin connection drifts are then developed across the target orebody.

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 depends 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.

Figure 16.26, Figure 16.27, Figure 16.28, Figure 16.29, and Figure 16.30 show the required mine development layout to enable the drift-and-fill mining method for each deposit.

Figure 16.26 Kakula Mine Development Framework

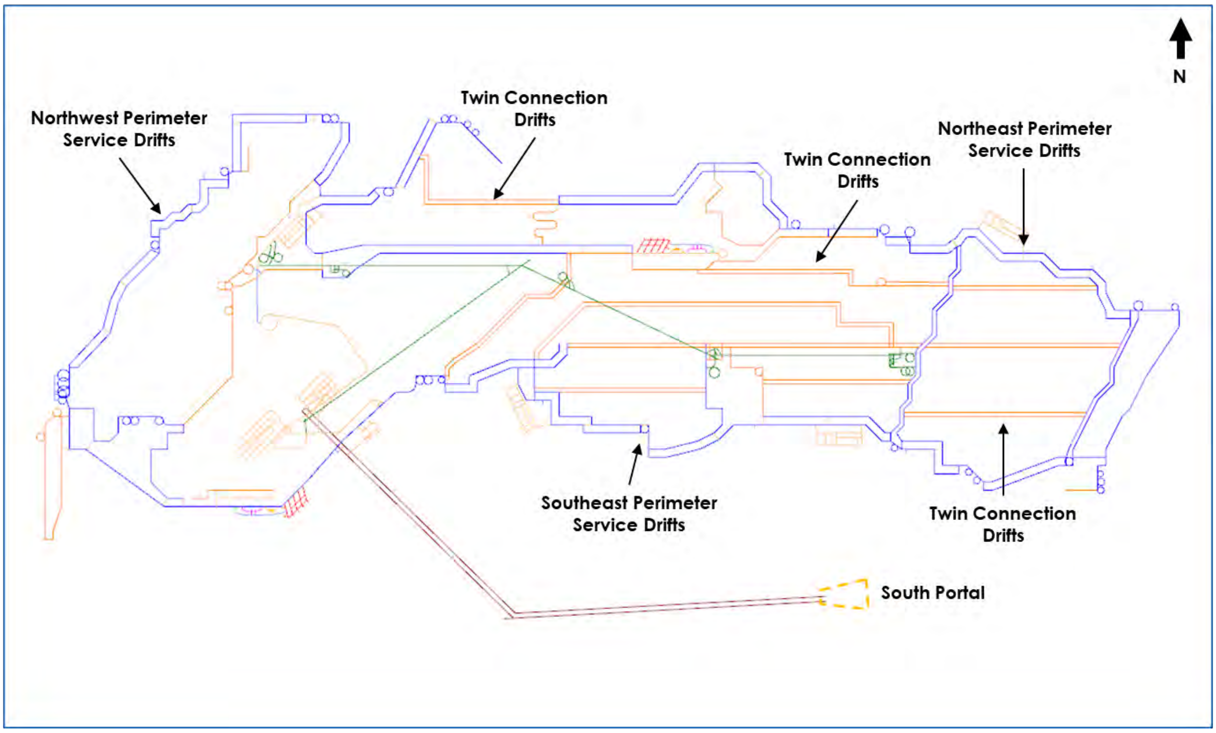


OreWin, 2023.

To establish the drift-and-fill mining method at Kakula, 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 extremities on the north and south side of the deposit. A single perimeter decline is developed to the west extremity on the south side of the deposit. Twin connection drifts are then developed across the targeted ore body, as shown in Figure 16.26.

It is important to note that as Kakula is an operating mine, some of the development and production has already been completed as shown in grey (as-built June 2022) in Figure 16.26.

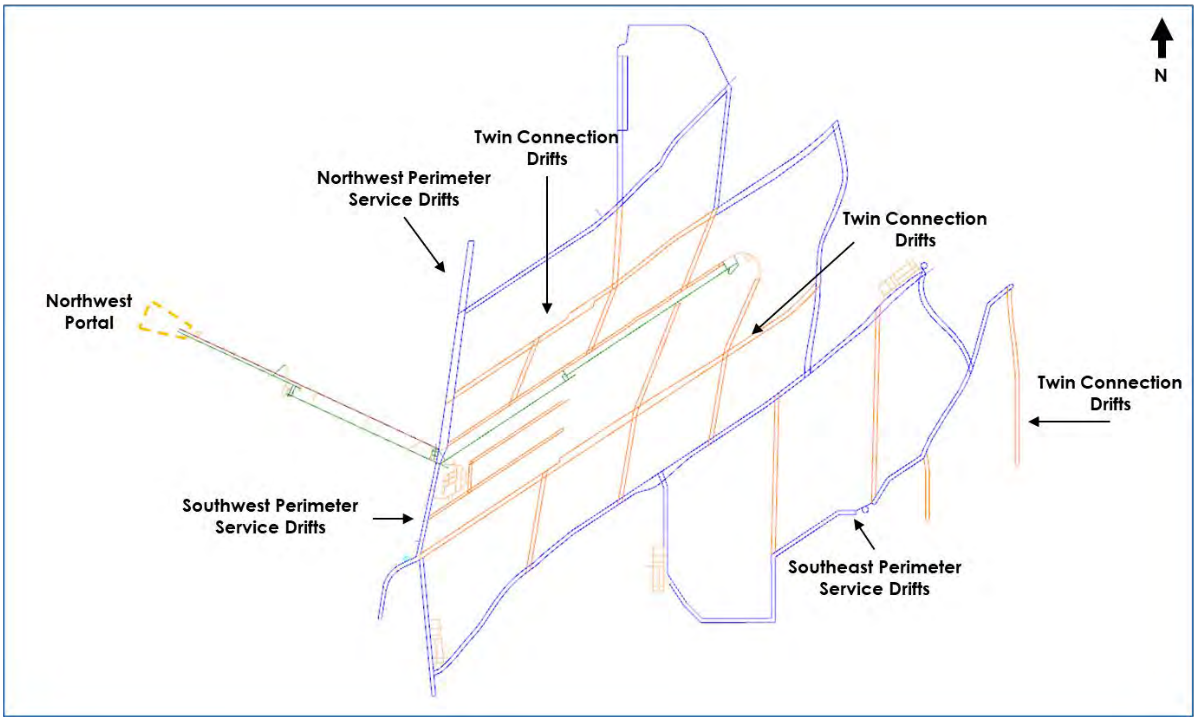
Figure 16.27 Kakula West Mine Development Framework



OreWin, 2023.

To establish the drift-and-fill mining method at Kakula West, a twin decline on the south of the targeted resource is driven at a defined offset. The perimeter drifts are then developed to the east extremities on the north and south side of the deposit. A single perimeter decline is developed to the north-west extremity on the west side of the deposit. Twin connection drifts are then developed across the targeted ore body, as shown in Figure 16.27.

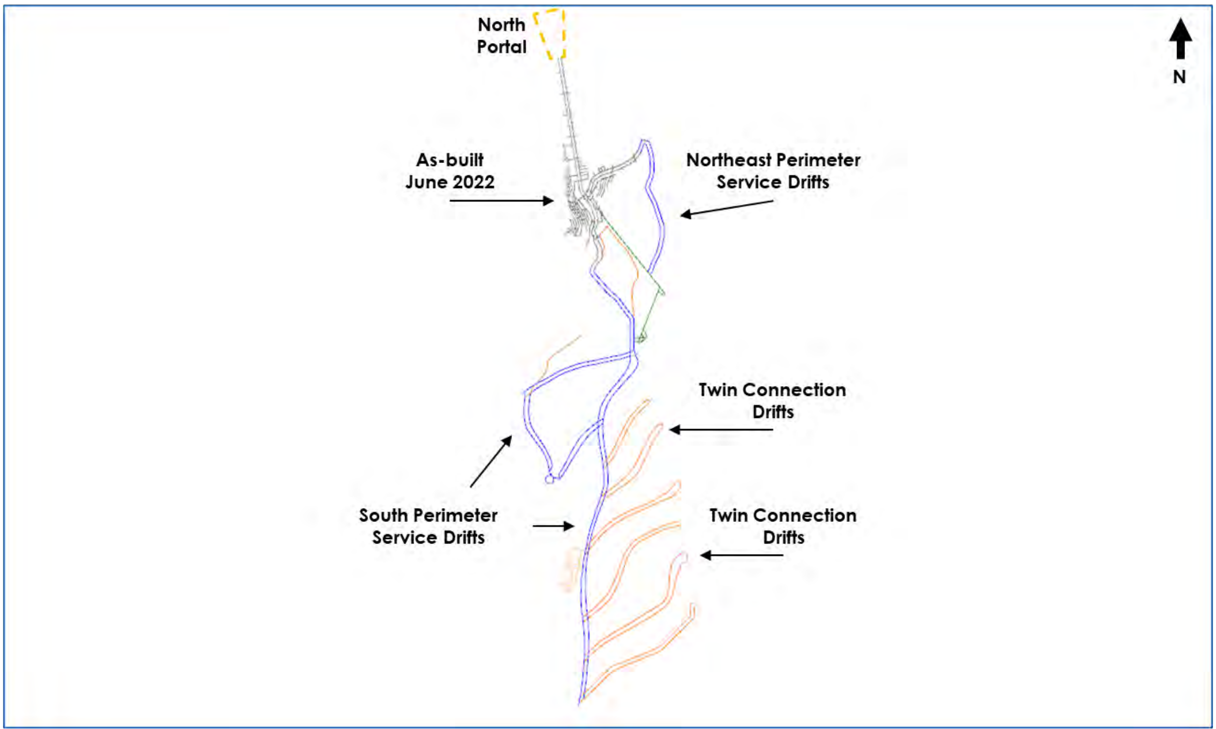
Figure 16.28 Kamoa 1 Mine Development Framework



OreWin, 2023.

To establish the drift-and-fill mining method at Kamoa 1, a twin decline to the north-west of the targeted resource is driven at a defined offset. This decline will also service the Kamoa 2 deposit. Access to the Kamoa 2 deposit branches off at the midpoint of the decline. From the base of the twin decline, the perimeter drifts are first developed to the northern and southern extents of the orebody. The twin perimeter service drifts are then developed to the east extremities on the north and south side of the deposit. Twin connection drifts are then developed across the targeted ore body, as shown in Figure 16.28.

Figure 16.29 Kansoko Sud Mine Development Framework

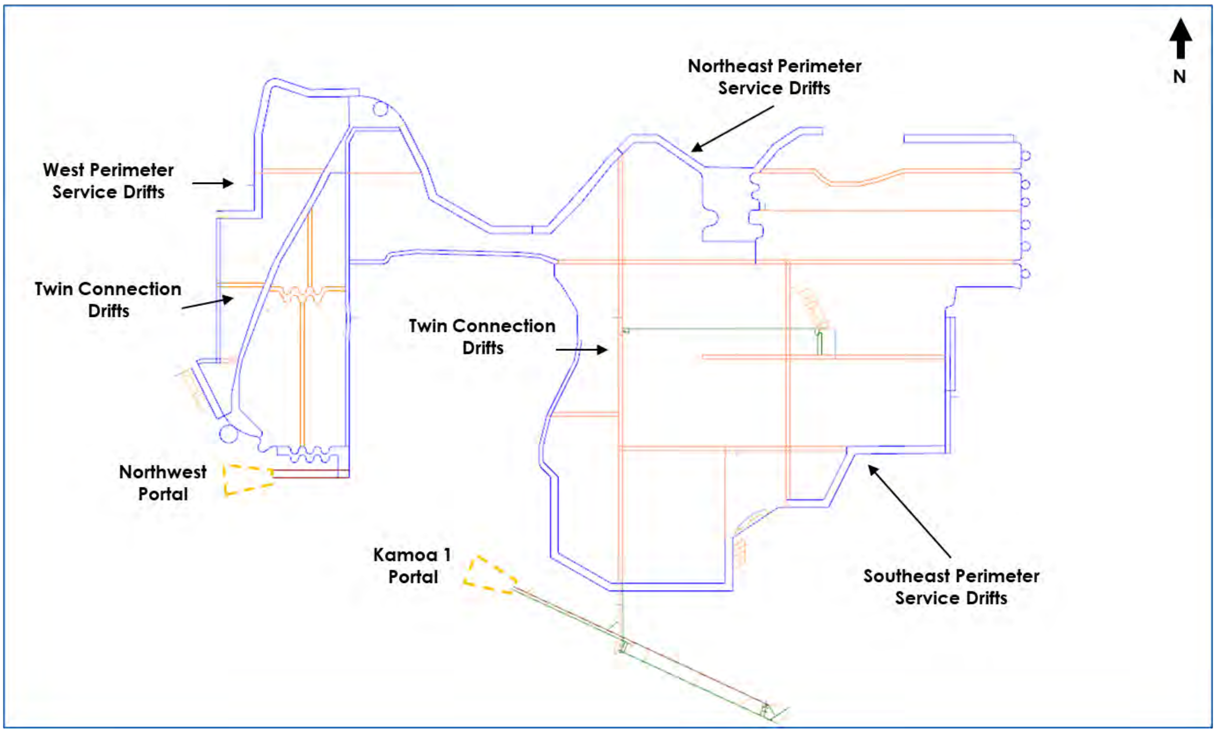


OreWin, 2023.

For Kansoko Sud's mine development framework, a twin decline to the north of the targeted resource is driven at a defined offset. From the base of the twin decline, the service drifts are developed through the middle of the orebody to the southern extents of the deposit. There are two sets of perimeter service drifts that enable access to the outer sections of the deposit, one in the north-east, and the other in the south-west as shown in Figure 16.29. Twin connection drifts are then developed across the targeted ore body, as shown in Figure 16.29.

As Kansoko Sud is an operating mine, some of the development and production has already been completed as shown in grey (as-built June 2022) in Figure 16.29.

Figure 16.30 Kamoa 2 Mine Development Framework



OreWin, 2023.

For Kamoa 2, a twin decline to the south of the targeted resource is driven from the previously established Kamoa 1 decline and will provide access to the eastern section of the deposit. An additional twin decline is driven from the north-west of the deposit and will provide access to the western section of Kamoa 2. From the base of the southern twin decline, the perimeter service drifts are then developed to the north-east and south-west extremities of the orebody. Similarly, twin perimeter service drifts are driven from the base of the north-west decline to the north-west extremities of the deposit. Twin connection drifts are then developed across the targeted ore body, as shown in Figure 16.30.

16.2.2 Drift-and-Fill

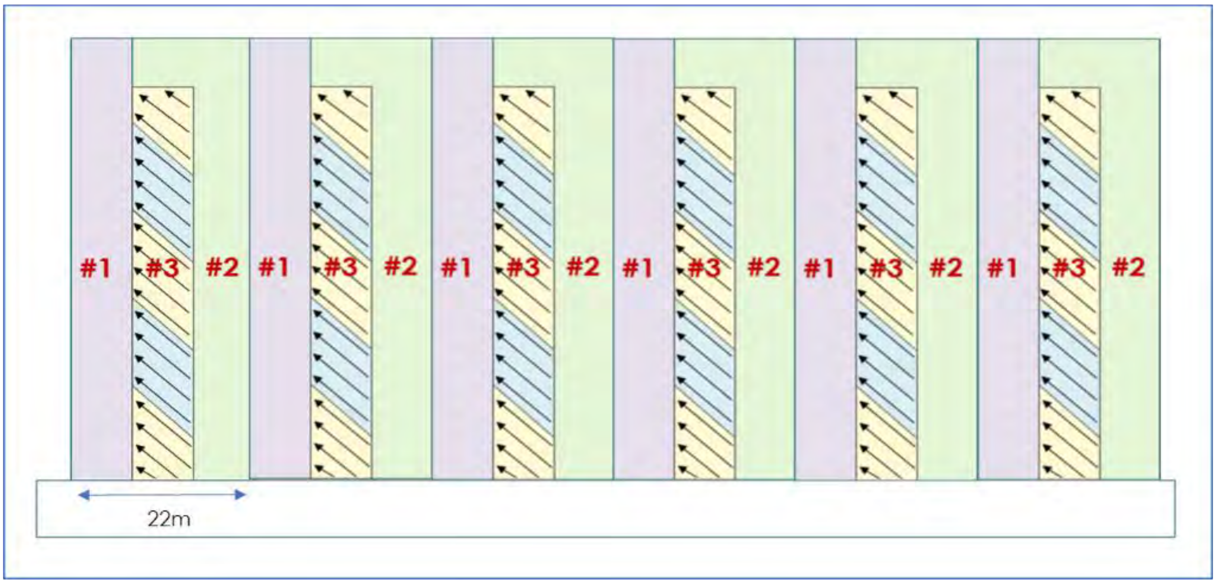
The chosen drift-and-fill method progresses by mining the first set of primary production drifts from the connection drifts. The primary production drifts are then immediately tight backfilled. The secondary production drifts can begin mining once the two primaries either side have been backfilled and cured. Once the secondary is completed, sliping can begin towards the access. Both the secondary and slipe drive are then backfilled and cured together. The panel progresses from left to right with the primary mining front ahead of the secondary/slipe sets. Each drive is supported using split sets at a set spacing, with the secondary drives supporting the slipe drives through the use of cable bolts, as per the geotechnical requirements in Section 16.1.4.

The dimensions of the primary, secondary, and sliping production drifts are 8 m W, 8 m W, and 6 m W respectively. Each set of production drifts have a varying height of 5.0–7.6 m depending on the thickness of the ore. A typical mining block with production drifts perpendicular to the connection drift is 198 m. Each block consists of nine primary headings, nine secondary headings, and nine slipe headings. Each mining block can be divided into three mining units, with three sets of production headings in each. There are no barrier pillars required between blocks.

The length of the 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.

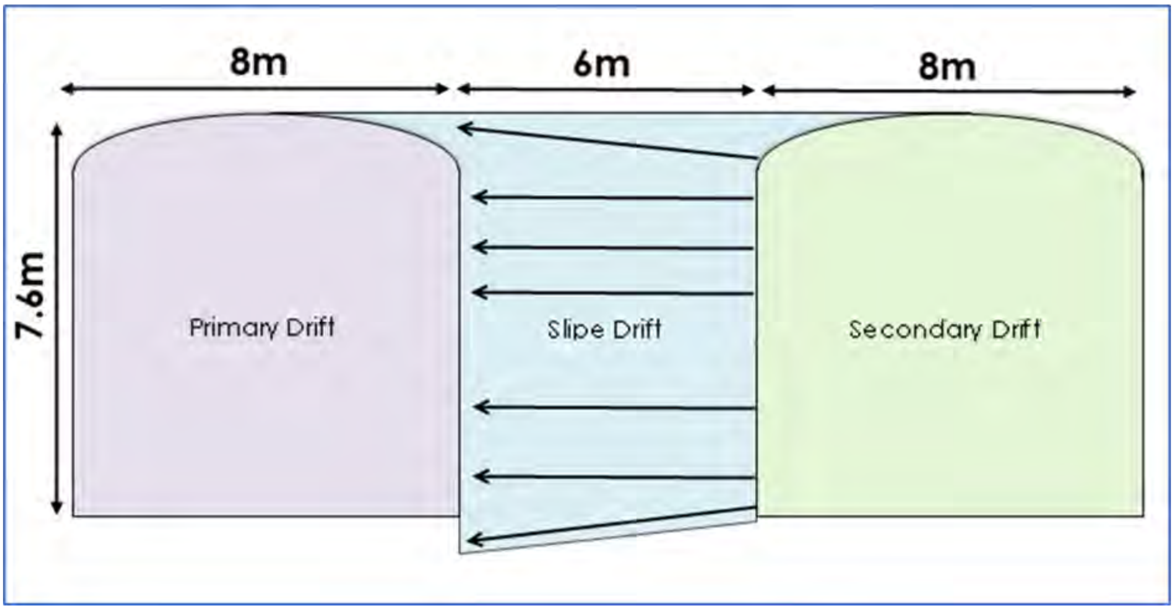
An example of the drift-and-fill mining method sequence can be seen in Figure 16.31. A typical cross-section can be seen in Figure 16.32.

Figure 16.31 Drift-and-Fill Mining Method (Plan View)



OreWin, 2023.

Figure 16.32 Typical Drift-and-Fill Cross-Section at 7.6 m H



OreWin, 2023.

16.2.2.1 Backfill

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 the primary drifts, paste fill walls are constructed at the entrance to the drift. As the secondary and slipe drifts are to be backfilled together, the paste fill walls are constructed at the entrance to the drifts once mining of the slipe drift is completed. 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.

The Backfill System arrangement and design is reported in Paterson & Cooke: KCSA Phase 3 PFS Backfill System Design Report.

16.2.2.2 Second Lift

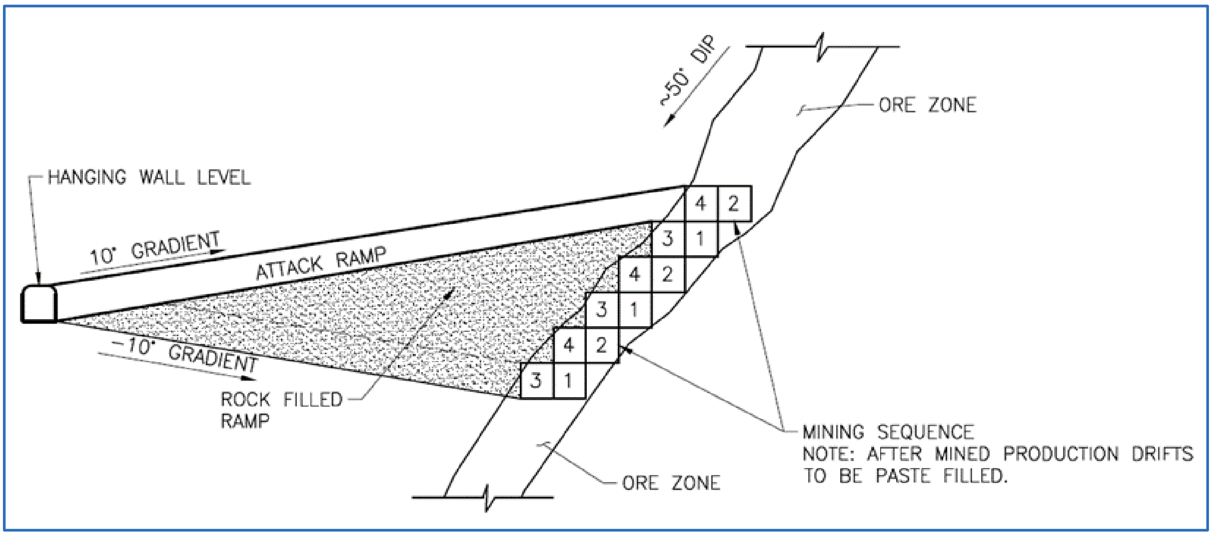
For ore thicknesses greater than 7.6 m, the first lift is mined at 7.6 m H, with the remaining ore then evaluated to determine if it is suitable for mining with a 3.0–7.6 m H second lift. The second lift production is accessed from a second lift connection drift, developed from the existing first lift 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. This arrangement satisfies geotechnical criteria and provides advantages in mining sequence, as well as better gains in ore production. The second lift can begin once the first lift has been extracted, backfilled, and cured. Sequence of extraction of the second lift follows the same process as the lift below. Ventilation loops are re-established by the new connection drives as they are connected to the perimeter declines.

16.2.2.3 Hanging Wall Accessed Drift-and-Fill (HWAD&F)

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

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 as shown in Figure 16.33. Two or less active drifts per attack ramp are to be mined at a given time.

Figure 16.33 Section View of Hanging Wall Accessed Drift-and-Fill Area



OreWin, 2023.

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.

16.2.3 Cut-off Grades

The following criteria were used to develop the cut-off grade strategy. For mine planning, the copper price used to calculate the NSRs in the block models was US$3.10/lb. Each ore type has a different recovery and could be processed through the smelter or sold as concentrate, this results in a different copper cut-off grade for each ore type. For consistency an elevated cut-off of US$100/t NSR was used to define the stoping shapes. A marginal cut-off of $80.00/t NSR was used for development. These were used at the start of the mine planning process to define stoping shapes which were then refined by the mine design work. Revisions and updates of the long-term holistic mine planning will result in changes to cut-off grade strategy as the Mineral Resources and Mineral Reserves are further defined. The analyses should include 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. To identify this potential, further study will be needed. These studies will be undertaken using a holistic approach into the long-term options to maximise the efficient extraction of the Kamoa-Kakula Mineral Resources. Cut-off grades can be expected to be lower than those used in Kamoa-Kakula IDP23 to achieve this.

The copper cut-off grades equivalent to $100/t NSR for sale of metal from the smelter or sale of concentrate are shown in Table 16.18.

Table 16.18 Copper Cut-off Grade Equivalents

Ore Type	Net Smelter Return ($/t NSR)	On-site Smelter (% Cu)	Concentrate Sales (% Cu)
Kakula	100	1.91	2.17
Kamoa Supergene	100	1.86	2.20
Kamoa Hypogene	100	1.88	2.34

16.2.4 Dilution

The following criteria were used to develop the dilution strategy.

16.2.4.1 Dilution Grade

All production drift dilution grades are calculated by interrogating the Mineral Resource block model. Dilution due to overbreak into surrounding waste rock or backfill has been applied using factors within the schedule. All production headings mined adjacent to backfilled headings have a paste dilution component calculated within the schedule to produce final diluted grades. The paste fill dilution tonnage has a zero-grade copper value.

16.2.4.2 Dilution Tonnes

Drift-and-Fill

Drift-and-fill will be the primary method of ore extraction. Total ore tonnes above the cut-off grade can include the connection drifts, production access drifts, and the primary, secondary, and slipe headings in the production blocks.

All drift-and-fill production headings have a dilution factor applied due to overbreak into adjacent backfilled headings. The total actual dilution applied is approximately 8.6–11.6%. Primary drifts have no dilution factor applied as any overbreak into the walls is within the planned excavation. It is assumed that secondary drifts have 0.1 m of overbreak into the adjacent backfilled primary drift as shown in Figure 16.34. Overbreak into the unmined slipe drift is in ore, and not included as dilution. For each slipe drift it is assumed that 0.3 m of overbreak occurs into the adjacent backfilled primary drift. The overbreak backfill tonnage has a zero-grade copper value. An overall dilution factor can be determined based on the ratio between backfill overbreak and production heading dimensions. As the drift-and-fill method is a repeating pattern, a dilution factor of 1.82% has been applied to all production shapes.

Figure 16.34 Typical Drift-and-Fill dilution due to overbreak into backfill

OreWin, 2023.

It is important to note that the Kakula dilution factors applied to production shapes differ from the other PFS deposits based on discussion with the operating site team. The applied Kakula dilution factors are higher in the initial years (Years 1–2) and decrease as time progresses. Table 16.19 shows how the production shape dilution factor applied changes with time.

Table 16.19 Kakula Production Shape Dilution Factor Applied

Kakula Production Shape Dilution Factor Applied	
Description	Factor Applied
Years 1–2	5%
Years 3–5	4%
Years 6+	2%

Further to the 2-5% dilution applied from overbreak in the walls approximately 6.6% dilution is indirectly applied from the inclusion of material below cut-off in the floor of the headings (approx. 0.5 m). The inclusion of this material in the floor of the heading is due to the limitations in the granularity of block model. Therefore, total actual dilution applied is approximately 8.6-11.6%.

16.2.4.3 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; 0.1 m of rock material is lost on the floor. The mining method selected (drift-and-fill) for the Kamoa-Kakula deposits are development intensive and have recoveries similar to ore development. The average height of the material lost is estimated to be 0.1 m where the mining is single lift.

As Kakula is a currently operating mine, the existing as-built development and production headings were mined using a different methodology, and hence, its recovery factor has been lowered from 98.4% to 75%. The lower recovery factor for this material will take into account the losses associated with retreat mining at the end of mine life.

16.3 Kakula Underground Mining

16.3.1 Introduction

Based on updated design criteria, the mining method, mine design, and production schedule have been updated from previous studies. Mining method 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 selected and the resultant mine designs and schedules.

The mining methods used for the Kakula deposit are drift-and-fill and hanging wall access drift-and-fill. The Kakula Mineral Reserve by mining method is summarised in Table 16.20.

Table 16.20 Kakula Mine Probable Mineral Reserves by Mining Method

Production by Mining Method	Ore (Mt)	TCu (%)	Fe (%)	As (%)	S (%)
Ore Development	6.70	4.75	4.90	0.00	1.31
Drift-and-Fill	117.28	4.95	4.83	0.00	1.35
HWAD&F	14.23	3.47	5.11	0.00	1.26
Total Ore*	138.21	4.79	4.86	0.00	1.34

*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/t NSR (net smelter return). The ore zone density has been defined as using a greater than 2.4% Cu (total copper grade) cut-off. The swell factor for development is 50%.

Table 16.21 details the bulk density parameters of the ore and surrounding waste rock of the Kakula deposit.

Table 16.21 Bulk Density/In Situ by Area

Bulk Density/In Situ	Min (t/m^3)	Max (t/m^3)	Average (t/m^3)
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 and south declines have a maximum gradient of ±9°. The conveyor drift is 7.5 m W x 6.0 m H with 1.5 m arch corners, and 6 m W x 6.0 m H with 1.5 m arch corners for the service drift. 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 ±9°. They are 7.5 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 crosscuts 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° and 1.5 m arch corners.

16.3.3.2 Vertical Development

Vertical development consists of ventilation raises, bins, and boreholes. All ventilation raises are excavated with a raisebore drill. All ventilation shafts are designed at 6 m in diameter. Twin access drifts to the ventilation shafts are 7.5 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 Drift-and-Fill

The majority of mining at Kakula utilises the drift-and-fill mining method. The method is explained in detail in Section 16.2.2.

A typical mining block with production drifts perpendicular to the connection drift is 198 m wide. Each mining block consist of three mining units. Each mining unit comprises three primary headings, three secondary headings, and three tertiary headings.

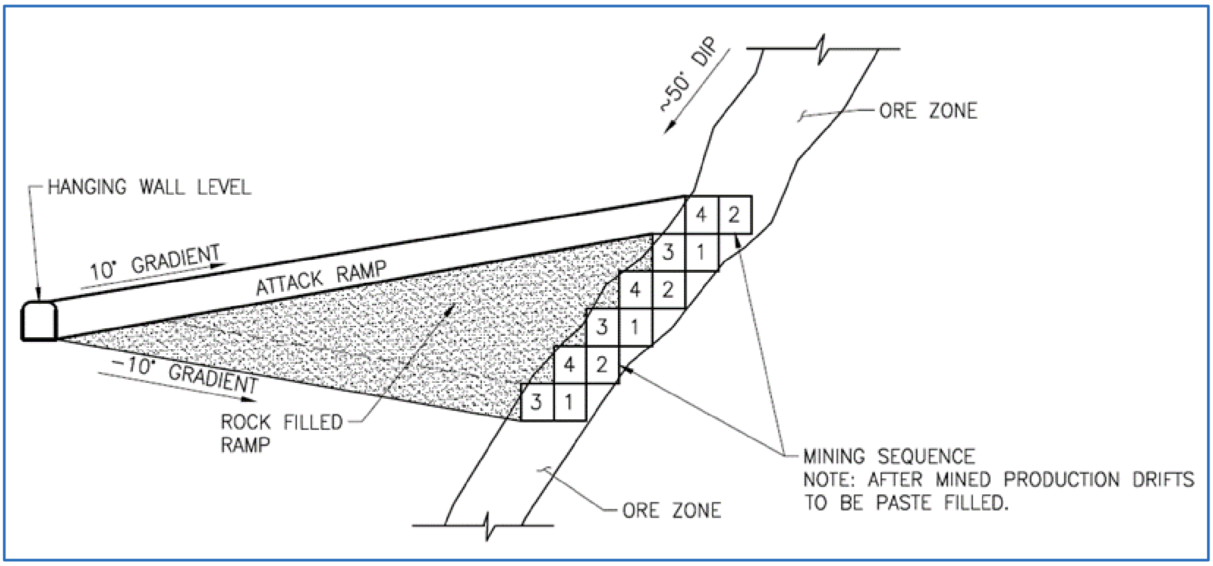
The production drift cross-sectional shape differs with the type of production heading. Typical primary and secondary productions drifts are an 8 m wide arch (1.5 m arch corners), with a maximum height up to 7.6 m. Slipe drifts are a 6 m wide arch (1.5 m arch corners), with a maximum height up to 7.6 m.

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 (9°).

16.3.3.4 Hanging Wall Accessed Drift-and-Fill (HWAD&F)

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 HWAD&F method is explained in more detail in Section 16.2.2.3. 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 16.35.

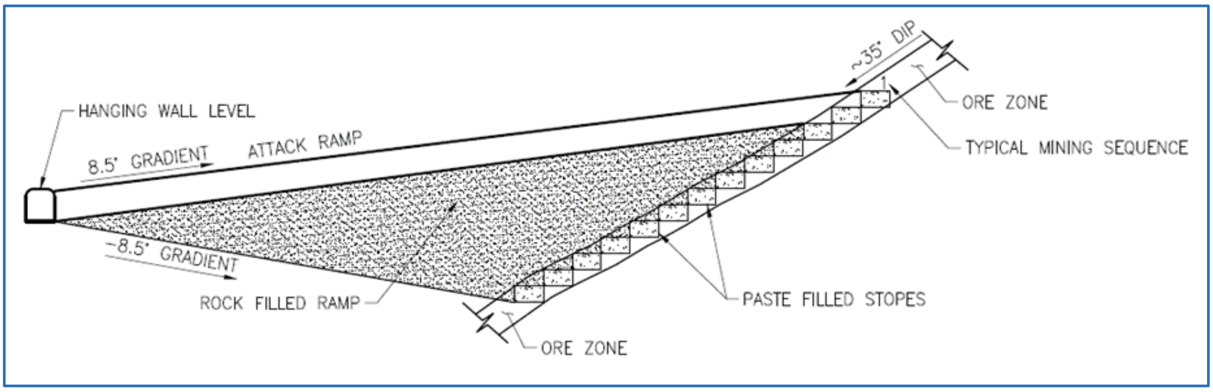
Figure 16.35 Section View of West Hanging Wall Accessed Drift-and-fill Area



OreWin, 2023.

The East area production drift level comprises of 12 horizons of production drifts measuring 3.0 m x 6.0 m W, as shown in Figure 16.36. Production drifts are accessed through attack ramps from the level development and developed on strike.

Figure 16.36 Section View of East Hanging Wall Access Drift-and-Fill Area



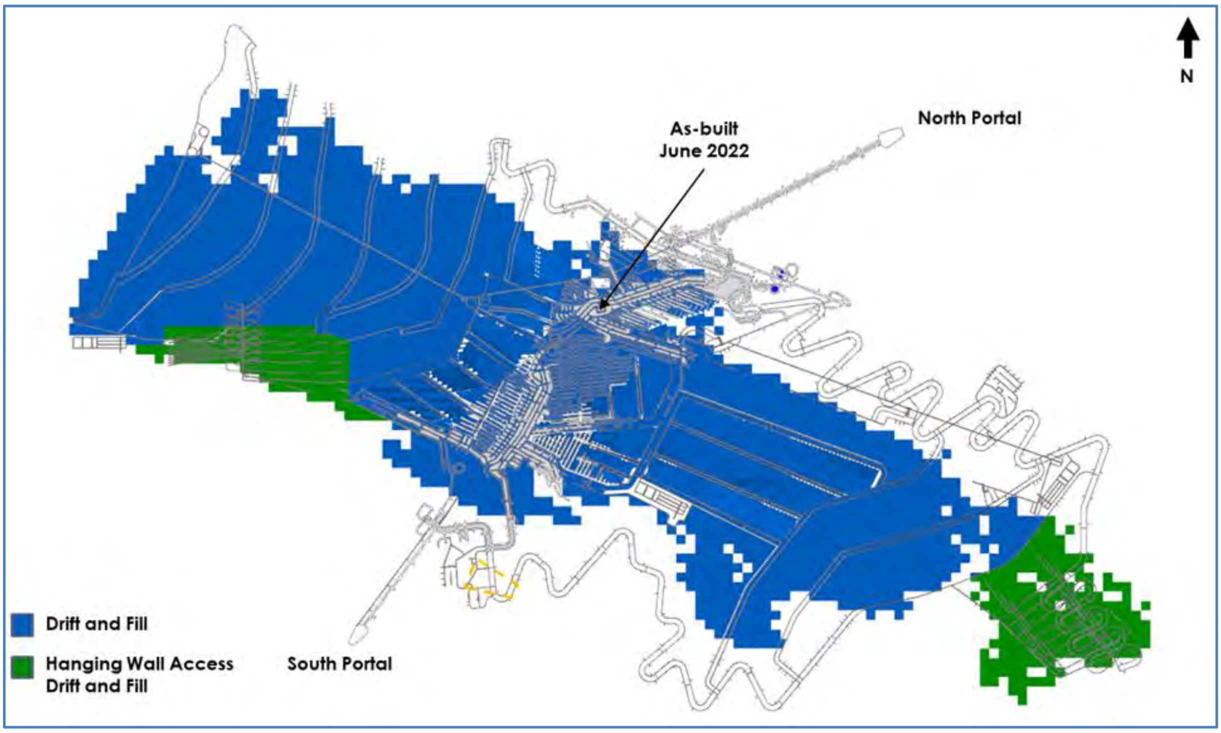
OreWin, 2023.

16.3.3.5 Mining Method Selection

The primary mining method for the Kamoa-Kakula deposits (drift-and-fill) was selected based on the results of a trade-off study which compared six different variations of drift-and-fill mining to identify the most suitable.

The main production areas of the deposit will be mined using variations of the drift-and-fill method. Identified mining areas with a dip greater than 25° (about 11% of mining areas) will be mined using the hanging wall access drift-and-fill (HWAD&F) method. The remainder will be mined using the drift-and-fill method as shown in Section 16.2.2. The Kakula deposit is illustrated in Figure 16.37, represented by mining method.

Figure 16.37 Kakula Mine Mining Method by Location



OreWin, 2023.

16.3.4 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.

16.3.4.1 Backfill Schedule

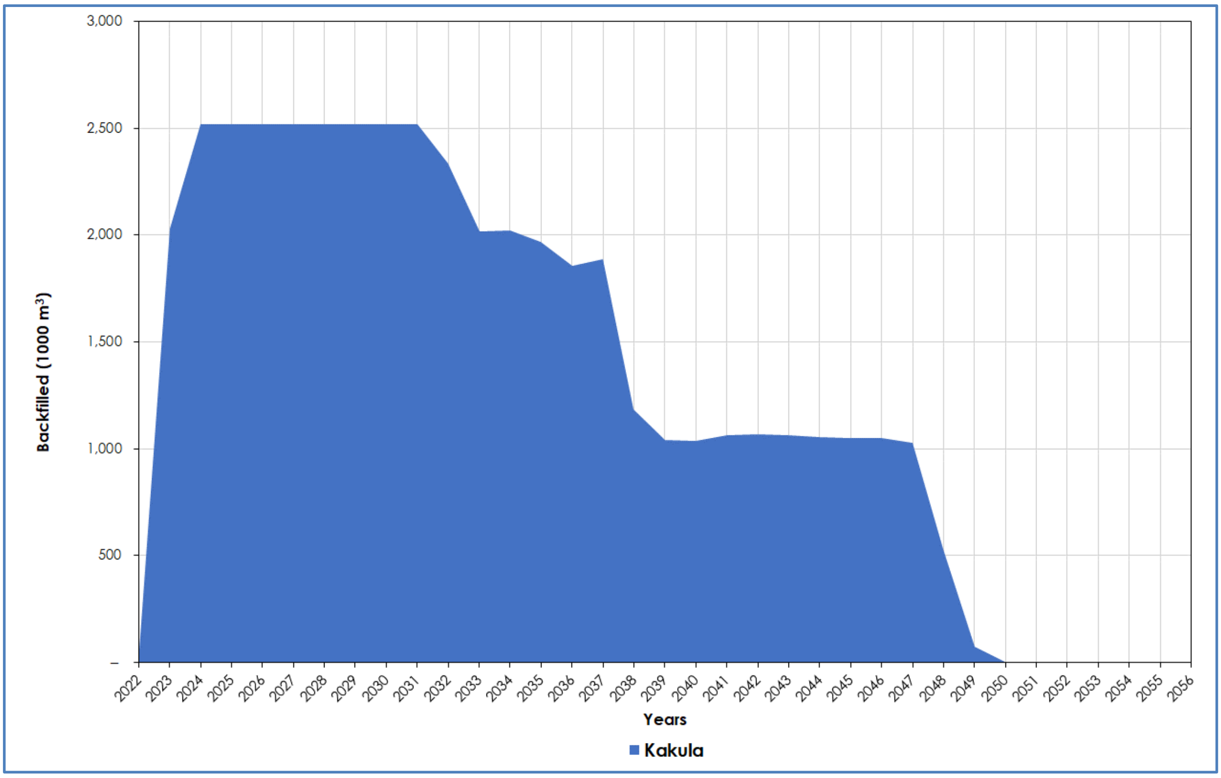
The underground paste distribution system is based on the life-of-mine requirements of the drift-and-fill mining.

Table 16.22 and Figure 16.38 detail the fill requirements by year.

Table 16.22 Kakula Mine Fill Requirements by Year

Cemented paste fill ('000 m^3)									
Year	2023	2024	2025	2026	2027	2028	2029	2030	2031
'000 m^3	2,030	2,520	2,520	2,520	2,520	2,520	2,520	2,520	2,520
Year	2032	2033	2034	2035	2036	2037	2038	2039	2040
'000 m^3	2,336	2,016	2,021	1,969	1,855	1,887	1,183	1,042	1,035
Year	2041	2042	2043	2044	2045	2046	2047	2048	2049
'000 m^3	1,064	1,066	1,062	1,052	1,051	1,050	1,026	530	72

Figure 16.38 Kakula Mine Backfill Schedule



OreWin, 2023.

16.3.5 Mine Access Design

Box-Cuts

There are two box-cuts developed for access to the underground workings. Both the northern, and southern box-cuts incorporate two portals.

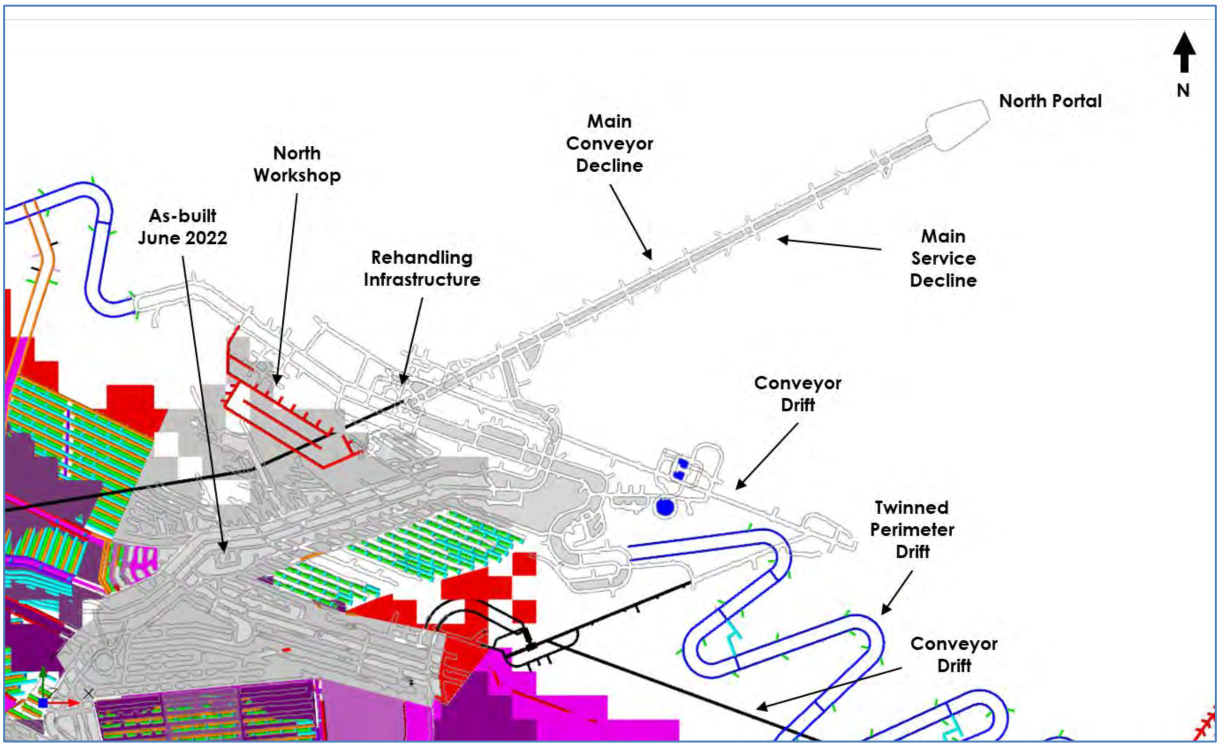
Main Declines

The deposit is accessed via twinned declines on the north and south side respectively.

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 6.0 m W x 6.0 m H, with the conveyor decline 7.5 m W x 6.0 m H. Both northern declines have a maximum gradient of ±9° gradient. Development of the declines has been completed as shown in Figure 16.39.

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 shown in Figure 16.39.

Figure 16.39 Kakula Mine Main North Access Development

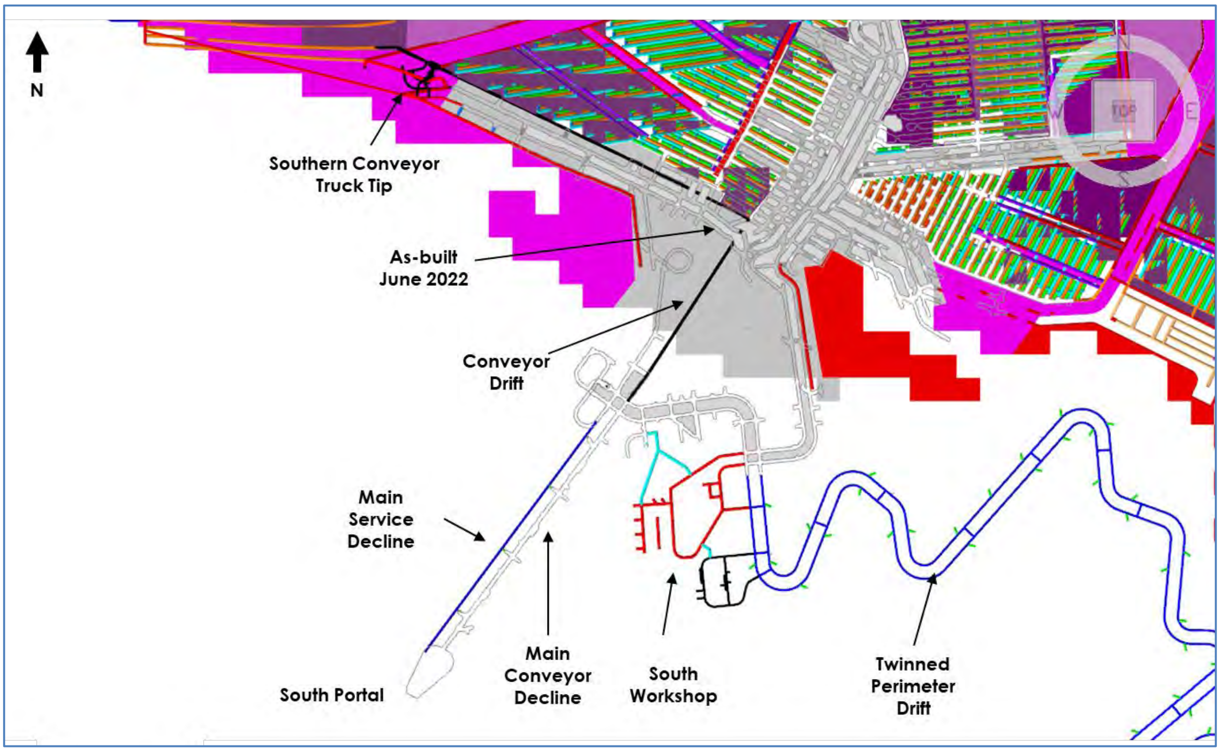


OreWin, 2023.

Similar to the north decline, the south decline has a primary mine service access, with the other a conveyor haulage drift. The service decline has dimensions of 6.0 m W x 6.0 m H, and the conveyor decline is 7.5 m W x 6.0 m H. Both southern declines have a maximum gradient of ±9° gradient. Development of the access declines has been completed as shown in Figure 16.40.

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 640 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, as shown in Figure 16.40.

Figure 16.40 Kakula Mine South Access Development



OreWin, 2023.

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 conveyor drifts are split into north and south systems, with the option to create a join if required. The northern conveyor systems extend to the lower north-western and north-eastern extremities of the orebody and converge at the northern main conveyor decline. The southern system extends to the upper south-western limit and conveys ore to surface via the southern main conveyor decline. Conveyor drifts are 7.5 m W x 6.0 m H and are driven at a maximum gradient of ±9.0°. Section 16.3.8.4 covers the rock handling systems in greater detail.

Mining Areas

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.

16.3.6 Mine Development and Production Schedules

As Kakula is a current operating mine, the development schedule focuses on continuing to establish mine services and supporting infrastructure to enable the production mining areas to ramp-up to 9.2 Mtpa ore production.

Table 16.23 summarises the LOM development and production results.

Table 16.23 Kakula Mine Life-of-Mine Development and Production Summary

Waste Development	
Lateral (m)	74,526
Lateral (kt)	7,215
Vertical (m)	16,766
Vertical (kt)	312
Production by Mining Method	
Ore Development (m)	53,598
Ore Development (kt)	6,699
Drift-and-Fill (kt)	117,282
HWAD&F (kt)	14,227
Total Ore Production	
Total Ore Development (kt)	6,699
Total Production (kt)	131,509
Total Tonnes (kt)	138,207
Diluted Grade	
TCu (%)	4.79
S (%)	1.34
As (%)	0.0001
Fe (%)	4.86

1. Notes: Vertical development includes boreholes.
2. Stope shapes designed on an NSR cut-off value of US$100/t NSR.

The following conditions were used in developing the LOM schedule:

- Proximity to the main accesses and initial 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.

16.3.6.1 Productivity Rates

Effective Operating Hours

The effective operating hours per shift are summarised in Table 16.24 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 16.24 Kakula Mine Shift Rotations and Effective Operating Hours Calculations

Shift Cycle	Calculations
Days per Year	360 days
Number of Crews in Rotation	3
Shifts per Day	2
Shift Duration	12 h
Travelling Time – In	19.50 min / 0.32 h
Travelling 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) and compared with inputs. The cycles were updated accordingly following team discussions. Mine productivities and schedule are based on a development rate of 120 m/month for all primary development.

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

The production rate in ore tonnes per month is highly variable as it is an outcome of the estimated development advance rate, combined with ore thickness, dilution parameters, and ore density.

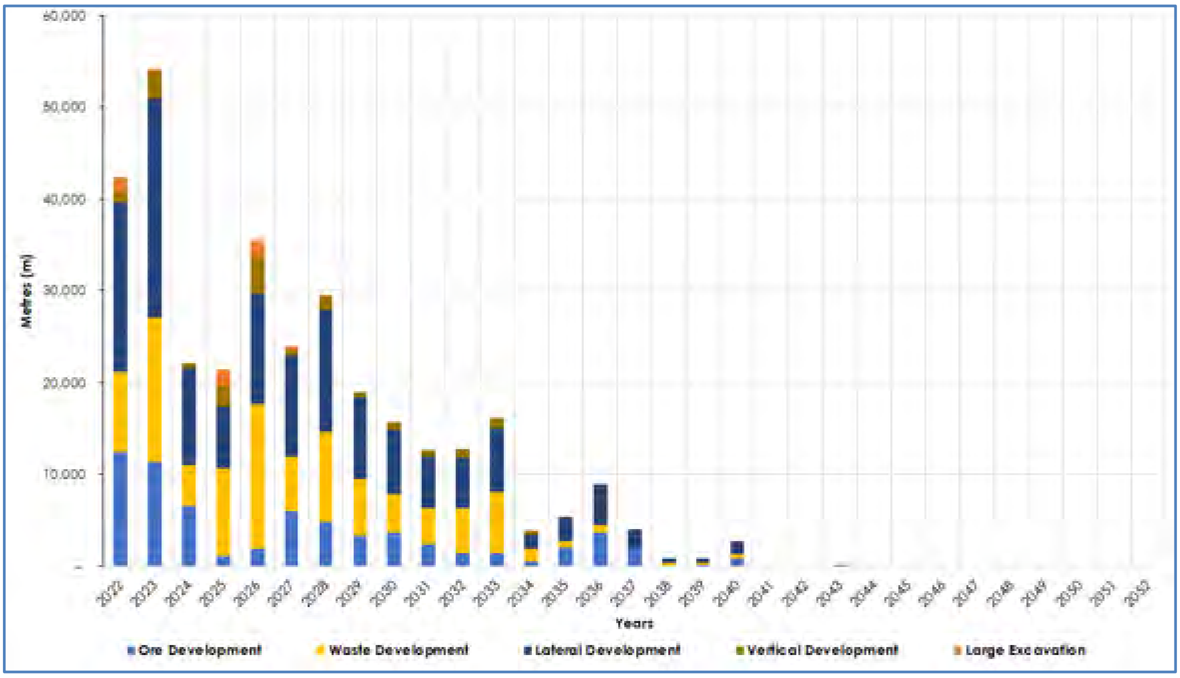
To determine the drift-and-fill production rates, development rates for primary, secondary, and tertiary drifts were combined with paste filling, cable bolting, and end-of-shift blasting restrictions in a block configuration, to determine the net block production rate for use in the schedule. The net block production rate was then applied to the drift-and-fill mining shapes within the schedule. The production rate was adjusted depending on the height of the production heading in order to better represent the change in condition. Paste fill barricade construction/placement as well as installation of split sets (geotechnical supports) was not considered in the cycle calculations as it will be completed off critical task.

16.3.6.2 Development Schedules

Development schedule

The life-of-mine development schedule targets the areas required to support the LOM plan. This includes excavating the perimeter service drifts, conveyor drifts, and key infrastructure associated with truck tips, ventilation, dewatering, and maintenance facilities in advance of production areas. Figure 16.41 illustrates the development metres associated with the LOM activities.

Figure 16.41 Kakula Mine Life-of-Mine Development Schedule



OreWin, 2023.

16.3.7 Mine Production Plan and Scheduling

The Kakula LOM milling schedule targets 9.2 Mtpa processed. The initial ramp-up scheduled tonnage will be supplemented with stockpiled ore in order to meet the processing target. Table 16.25 presents the annual ramp-up scheduled tonnes that were set to meet the 9.2 Mtpa production rate.

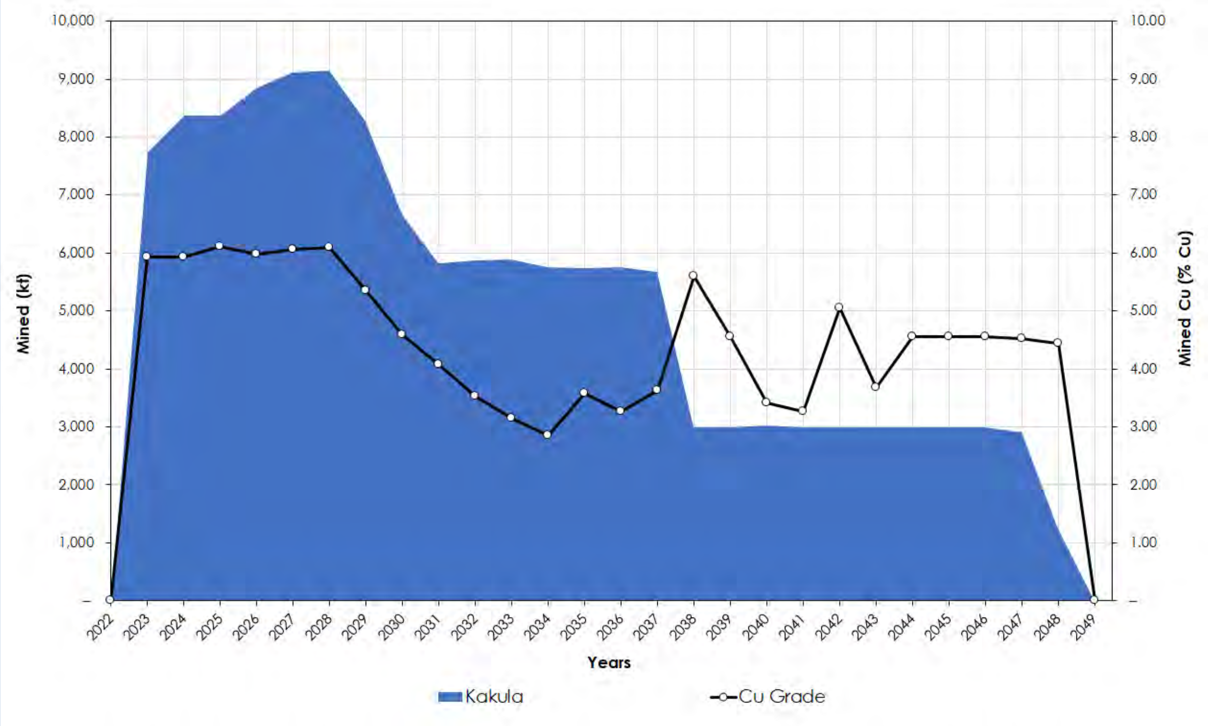
Table 16.25 Kakula Mine Ramp-Up Scheduled Tonnage

Production Schedule	Years	Scheduled Tonnes (kt)
Ramp-Up (2023)	1	7,736
Ramp-Up (2024)	1	8,370
Ramp-Up (2025)	1	8,365
Ramp-Up (2026)	1	8,858
Ramp-Up Total	4	33,329

Life-of-Mine Production Schedule

Full production of 9.2 Mtpa is sustained for two-years, starting in 2027, and tapering off in 2029. As Kakula West begins production (3.2 Mtpa), Kakula production will ramp down to 6.0 Mtpa steady state for nine-years (2029–2037), so that the total amount of material supplied to the Kakula mill remains at 9.2 Mtpa. As Kakula West ramps up to maximum production at 6.2 Mtpa, Kakula production will ramp-down to 3.0 Mtpa until LOM. The relationship between Kakula and Kakula West is continued for the Kakula LOM, so that both deposits ramp down to finish production together. The mining blocks are scheduled so that a higher TCu value is achieved early in the mine life. Figure 16.42 illustrates the LOM production schedule and copper grade.

Figure 16.42 Kakula Mine Life-of-Mine Production Schedule and Copper Grade



OreWin, 2023.

16.3.8 Underground Infrastructure

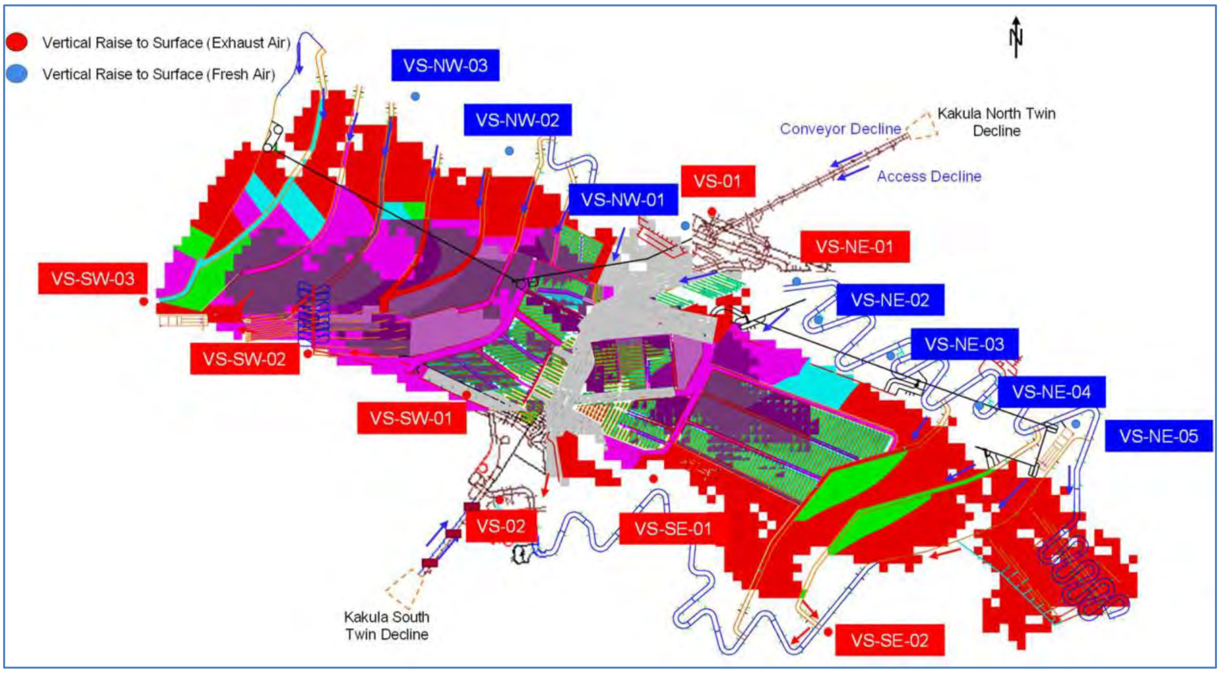
16.3.8.1 Mine Ventilation System – Kakula

The following assumptions were considered in the ventilation design to maintain safe operating conditions underground and to abide by applicable legislative requirements. Australian and 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.
- Primary and secondary leakage rates used for preliminary airflow estimates for when the mine is fully developed are 10% and 20% respectively. These factors are used 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 30 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.
- Heat load factors for diesel equipment have been split into four types. The following types of trucks, loaders, auxiliary and supporting equipment, have respective heat load factors of 1.1, 1.0, 0.8, and 0.3, applied as linear activity tracks in the Ventsim modelling.
- Diesel engines are assumed to have a power conversion efficiency of 35%.

The mine layout schematic illustrating the location of the ventilation shafts is shown in Figure 16.43. The ventilation system is designed to provide fresh air from the declines and through intake ventilation shafts (of which two will be equipped with coolers) located at the north of the orebody. The ventilation air will naturally flow through the perimeter drives connecting the north and south of the orebody. Air will return from south perimeter drives via eight ventilation shafts to surface located mostly on the south side of the orebody. VS-01, and VS-NE-01, will ventilate the conveyor belt directly to return. 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 16.43 Kakula Mine Layout with Ventilation Shaft Locations



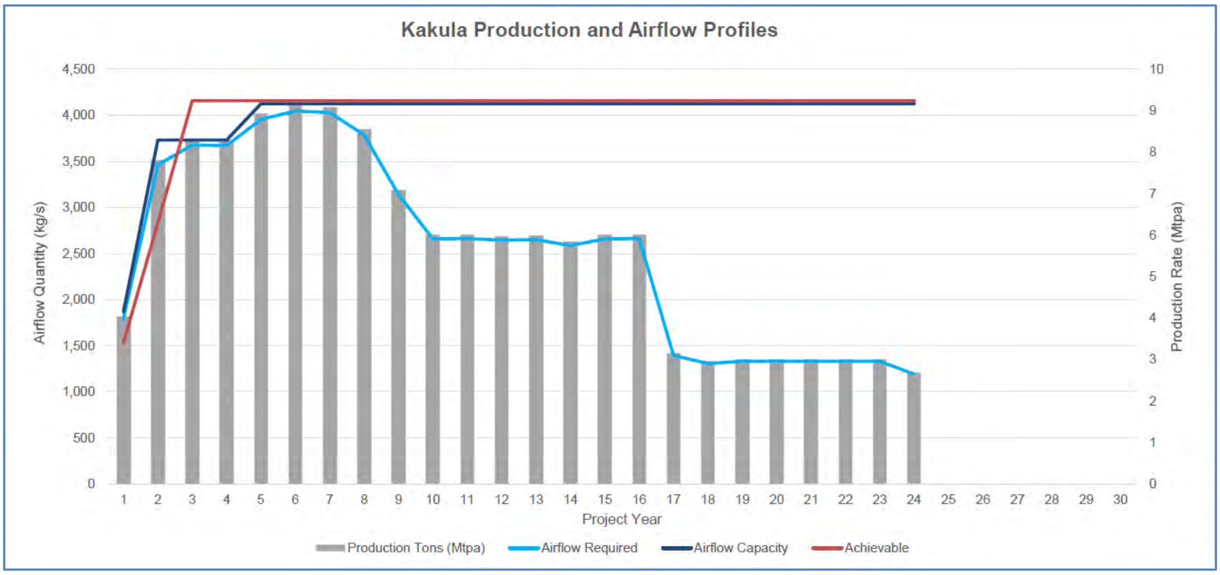
OreWin, 2023.

A summary of the primary ventilation fans is provided in Table 16.26. The LOM airflow requirements are shown in Figure 16.44. The model shows the primary ventilation requirements of 4,050 kg/s at the peak production rate of 9.2 Mtpa, and a maximum depth of 1,100 m.

Table 16.26 Kakula Mine Primary Main Ventilation Fan Requirements

Raise Location	No. of Fans (in parallel)	Operating Range (kg/s)	Peak Airflow (kg/s)	Peak Total Pressure at Collar (kPa)	Estimated Power (kW)
VS-1	2	220 - 440	220	1.2 – 1.5	380 - 1,100
VS-2	3	220 - 750	440	1.2 – 1.5	380 - 1,610
VS-NE1	2	220 - 440	220	1.5 – 2.2	470 - 1,385
VS-SE1	3	660 - 750	678	1.5 – 2.2	1,455 – 2,360
VS-SW1	3	660 - 750	678	1.5 – 2.2	1,455 – 2,360
VS-SE2	3	660 - 750	678	1.5 – 2.2	1,455 – 2,360
VS-SW2	3	660 - 750	678	1.5 – 2.2	1,455 – 2,360
VS-SW3	3	660 - 750	678	1.5 – 2.2	1,455 – 2,360

Figure 16.44 Kakula Mine Life-of-Mine Airflow Requirements



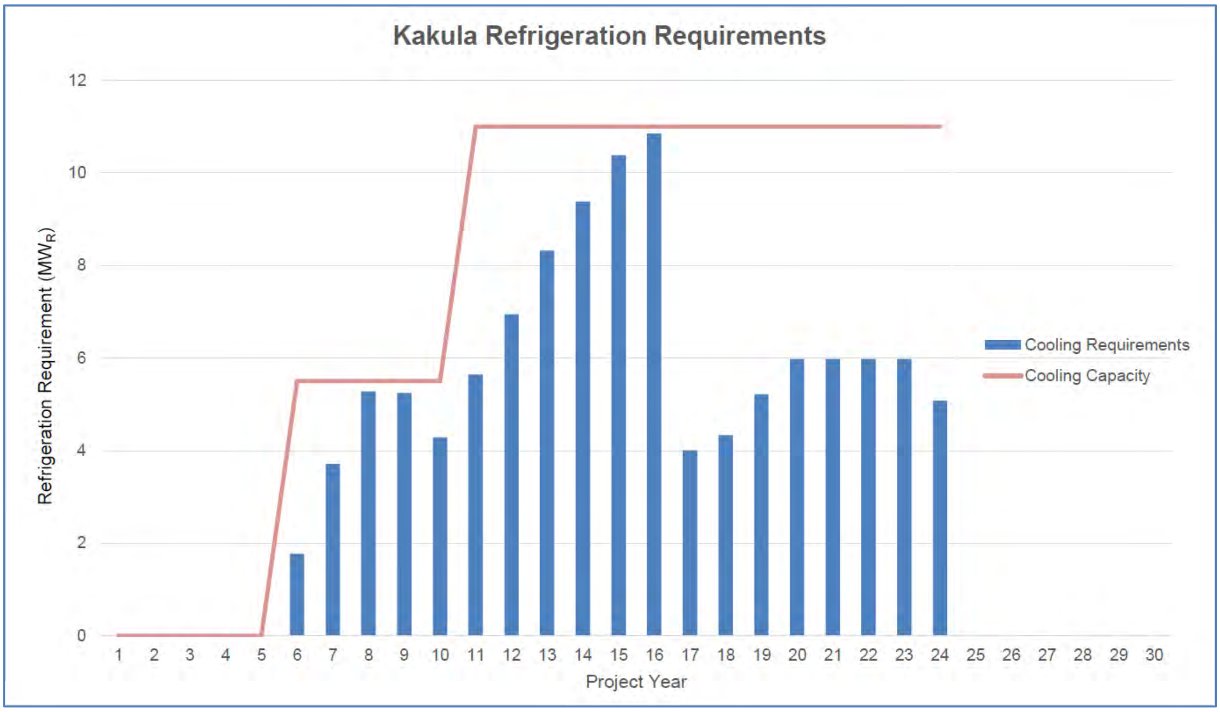
OreWin, 2023.

16.3.8.2 Mine Air Cooling Facilities – Kakula

The heat load and refrigeration requirements of the mine are shown in Figure 16.45. The cooling requirements were estimated from first principles and confirmed by VentSim models which considered auto-compression, surrounding and broken rock heat, vehicle heat, and all other heat components. The model indicated that 11 MW of cooling is required at the peak production rate of 9.2 Mtpa while producing up to 1,100 m below surface.

The refrigeration machines will be located in a refrigeration plant room near the Kakula North Twin Decline, serving two modular type BACs at the top of the new intake ventilation shafts (VSNE-02 and VS-NW-01). To provide a total cooling duty of 11 MWR (megawatts of refrigeration) to underground, 12 MWR of refrigeration machine duty will be required (including cooling losses). Half the refrigeration demand (6 MWR) will be required in PY6 and another 6 MWR in PY11. The refrigeration machines will be phased-in using 6 MWR and 5.5 MWR BACs.

Figure 16.45 Kakula Mine Life-of-Mine Refrigeration Load



OreWin, 2023.

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 16.27.

Table 16.27 Kakula Mine Auxiliary Ventilation Fan Requirements

Location	Flow per Fan (m^3/s)	Fan Total Pressure (Pa)	Fan Size diameter (m)	Duct Type	Estimated Power (kW)
Development Headings	36	2,300	1.40	Flexible	110
Drift-and-fill Headings	36	2,300	1.40	Rigid and Flexible	110

16.3.8.3 Mine Dewatering

Kamoa-Kakula 2023 PFS dewatering designs can be summarised into a Primary, and Secondary system, of which the Primary system comprises three typical, or standard pump station designs. The number of pump stations, design capacities and locations are based on the hydrogeological model that was developed by WSP Golders.

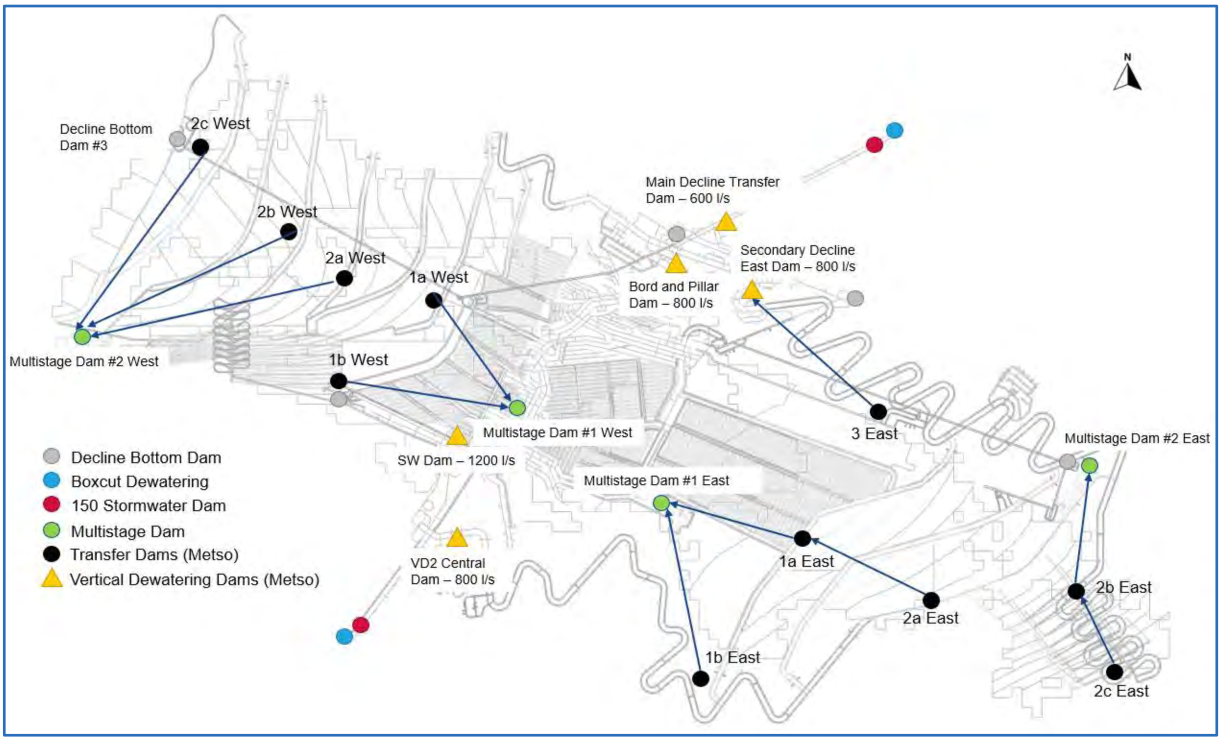
The Secondary pumping system extends from the face and feeds into the primary dewatering system. It comprises face pumps feeding into 2 m^3 portable skid pumps that feeds into larger 8 m^3 skid pumps. The number of skid pumps depends on the lift required to reach the primary dewatering system.

The Primary dewatering system comprises a combination of the following pump stations:

- Transfer dams: Centrifugal pumps installed five trains with one, two or three pumps in series, per train, depending on the dynamic head required. These pump stations cascade water to the dewatering pump stations or in certain cases out the decline to surface.
- Centrifugal dams: these have the same arrangement as the transfer pump station, but pump water to surface via boreholes.
- Multistage dams: these are large pump stations equipped with multistage pumps that can dewater the mine directly to surface from deeper parts in the mine. These dams are design with high flow rates in mind, and can dewater at rates of 1,500 l/s.

Current installed dewatering capacity at Kakula mine sits around 2,000 l/s, and the total is expected to increase to 3,200 l/s in future. The dewatering configuration for Kakula mine is illustrated in Figure 16.46.

Figure 16.46 Kakula Mine Dewatering Layout



DRA, 2022.

16.3.8.4 Rock Handling

Currently, rock handling at Kakula mine occurs from either the South, or the North. The Southern system relies on truck hauling through the VD#2 decline and overland, back to the Kakula stockpile. The Northern system features a rock handling system that includes two truck tips located at the decline bottom, a main decline conveyor, and a surface conveyor transfer system that can directly transport materials to the plant from underground or bypass them to the Kakula stockpile, which can be re-claimed via the bulk reclaim system.

As part of the Kamoa-Kakula 2023 PFS, there are plans for further expansion of the rock handling system both in the North and South. The Northern system will be extended both East and West. The Southern system, on the other hand, will be upgraded with the establishment of a new decline conveyor featuring a surface crusher, a bypass stockpile, and a bulk reclaim tip. The materials will then be transported via an overland conveyor to the ROM stockpile at the Kakula concentrator. These upgrades are deemed necessary to meet the required mining rate of 9 Mtpa.

Figure 16.47 illustrates the Kakula rock handling systems.

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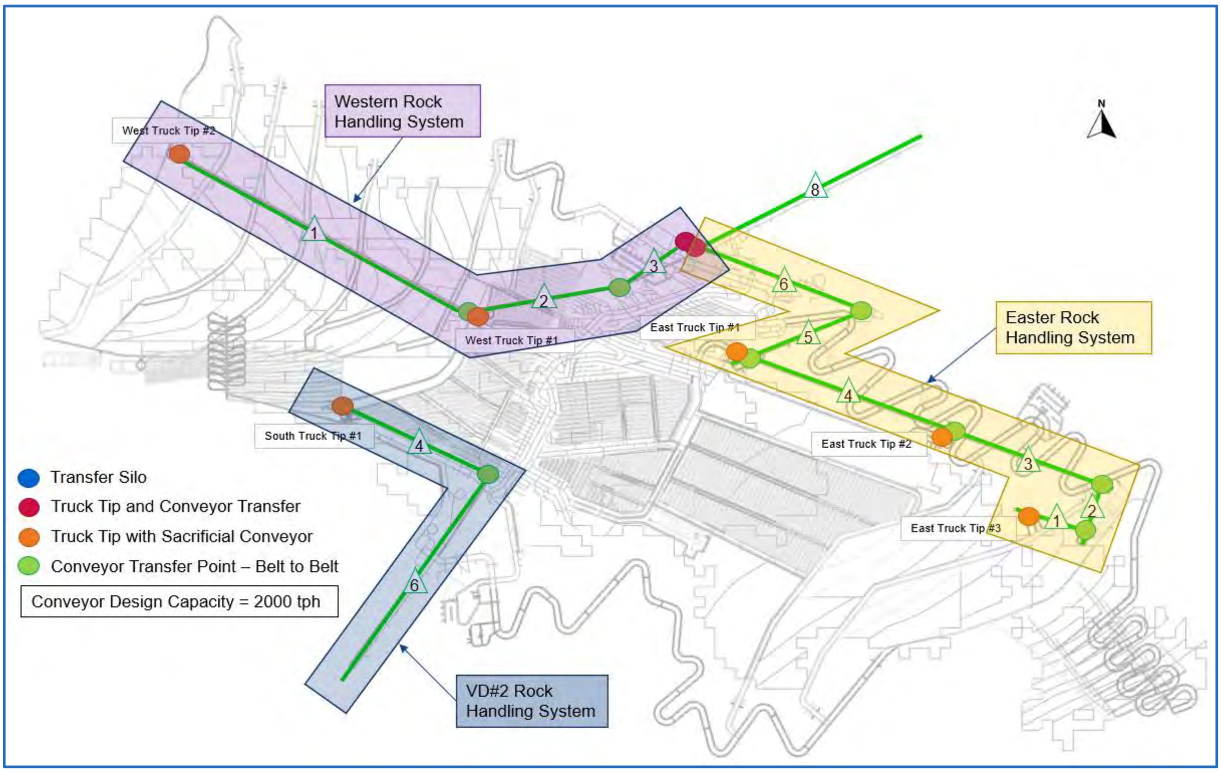
Figure 16.47 Kakula Mine Underground Rock Handling System Overview



DRA, 2022.

16.3.8.5 Materials Handling Logistics

The delivery, and distribution of materials at Kakula has been effectively established. Materials will be loaded onto LDV's, and UV's, at the surface stores, and transported underground via the service declines (North and South). Where feasible, materials will be stored underground in the vicinity of mining areas or workshops.

Emulsion is presently stored in emulsion tanks on the surface and then delivered underground through boreholes, equipped with pipes, to an underground storage tank, which will dispense it into Explosive UV's for underground use.

For shotcrete and other applications, concrete will be sent underground through a batch plant system that operates through boreholes. The cement mixture will be prepared on surface and delivered through a gravity feed down the pipe installed in the borehole underground, where it will be loaded into an agi truck for transportation underground.

Diesel bowsers are currently used to transport fuel from a surface storage facility located near the surface workshops. Once the underground workshop is established, boreholes will be drilled and fitted with pipes leading to underground storage tanks from which fuel, and lube, will be distributed by UV's underground. Refuelling stations will also be constructed underground near the workshops.

16.3.8.6 Workshops

Once driving up and out of the mine becomes unfeasible, major mobile equipment will remain underground for the duration of the machine's life cycle and will be serviced/ maintained in applicable underground workshops. Mobile equipment will only come out of the mine for a complete Original Equipment Manufacturer (OEM) rebuild, or to be scrapped and replaced.

The mining area will be serviced by a combination of main workshops and satellite workshops. The main underground workshops will be centrally located. As the mining progresses and travel distances increase, satellite workshops will be established near the production areas and furnished with the appropriate service equipment. Production fleet vehicles operating mainly at the production face (i.e., drill rigs and LHDs) will be serviced at satellite workshops. Minor repairs on this equipment will also be undertaken at the satellite workshops but will revert to the closest main workshop for major services and repairs.

Design Basis

Recommended maintenance interval information obtained from OEMs was used as a basis for the calculations to determine the number of workshops. Fleet sizes were estimated based off current and expected fleet sizing for each mine.

Main workshops have been allowed for which will be large workshops centrally located to service primary and secondary fleet. The main workshop will consist of multiple bays as described below.

- Wash Bays

Located at the entrance to the workshop for equipment to be cleaned prior to checking in for maintenance. Equipped with degreasers and manually operated high and medium-pressure sprayers supplied with service water. Oily water will be collected and cleaned with an oil water separator. Dirty oil will be collected for disposal in a used-oil IBC container, while separated water will be recovered through the underground dewatering system.

- Tyre Maintenance, Repair, and Storage Bay

This bay will be used to change wheel assemblies underground and will be equipped with a new and used tyre store for effective tyre management. Tyres will be handled using a telescopic material handler, and tyre handler attachment. The tyre bay will be equipped with overhead electrically operated crawls.

- Large Ramp Bays

These bays will be used to maintain large production equipment, like dump trucks, load haul dumpers, and drill rigs. They will be equipped with service ramps for chassis inspections and maintenance, as well as an overhead crawl beam for effective maintenance Figure 16.48.

- Large Crane Bays

These bays will be equipped with 25-tonne safe working load overhead travelling cranes for heavy lifting requirements during maintenance on large production equipment. It will also have sufficient headroom for full bowl extension underground dump trucks.

- Medium Service Bays

These bays will be used to maintain and repair medium-sized production and secondary equipment. They will be equipped with 10-tonne overhead travelling cranes for lifting requirements during maintenance Figure 16.49.

Figure 16.48 Large Ramp Bay

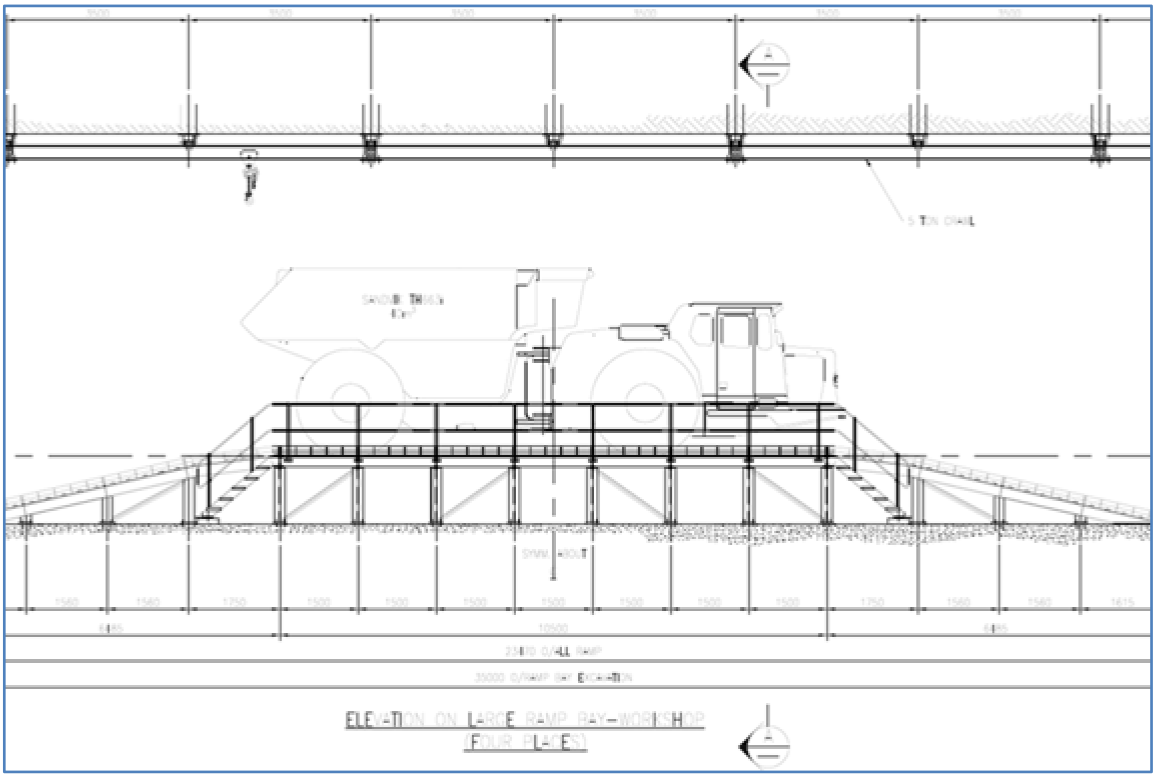
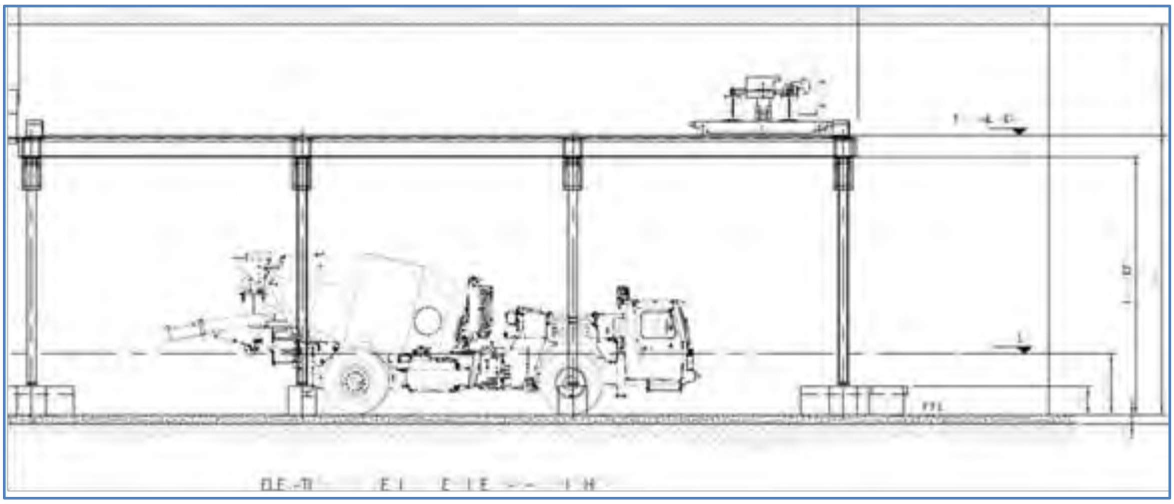


Figure 16.49 Medium Service Bay



The underground workshops will include supporting infrastructure, such as:

- Stores
- Waste collection
- Sub-assembly Stores
- Welding bays
- Offices, refuge bay, lunchroom, and toilet
- Refuelling station

Maintenance and Re-Build Philosophy

The underground workshops are designed to provide comprehensive maintenance and repair services for all primary, and secondary equipment, during scheduled, and unscheduled downtime. The execution of planned maintenance for all equipment is based on engine hours, as per the original equipment manufacturer (OEM) requirements. The replacement of major components is carried out underground, except for larger components such as boxes, buckets, and large axles, which are more easily handled on the surface. As an alternative to transporting heavy equipment to the OEM's workshop facilities, planned mid-life overhauls can be performed in the surface heavy workshops. The maintenance of Light Duty Vehicles (LDVs) is not included in the underground workshops' planned maintenance and will be performed at the surface LDV workshop.

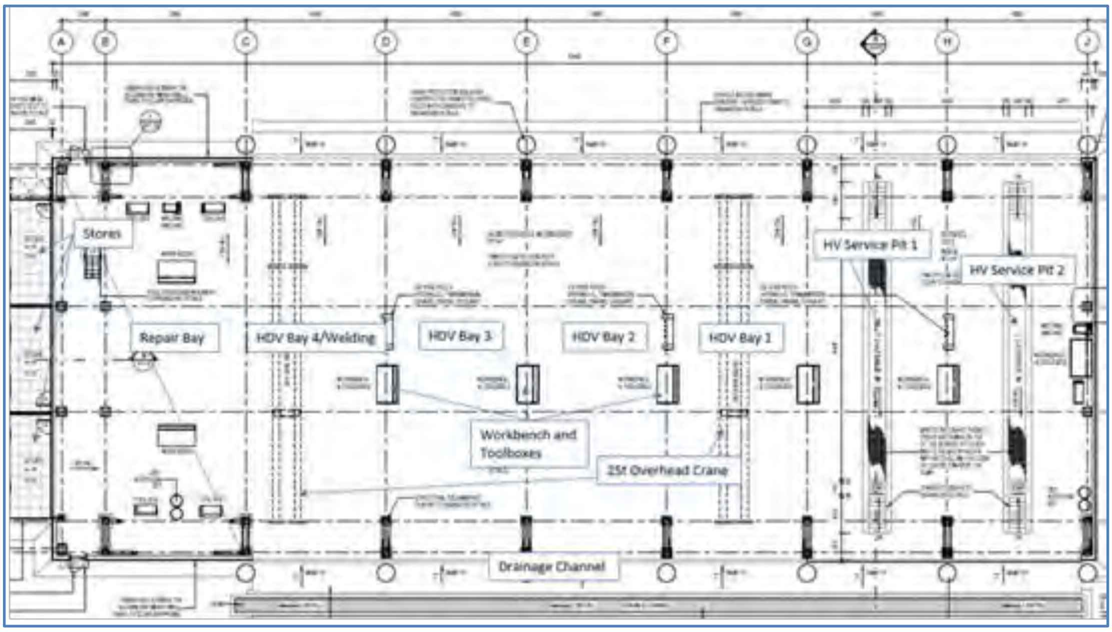
Satellite Workshops

Satellite workshops will be located near working sections and will be equipped to perform daily maintenance on slow-moving mobile equipment. The satellite workshops will have limited capacity and will have a three-bay design. The satellite workshops will contain a wash bay, service bay, and ramp bay. Concrete floors, lighting, hoisting arrangement, and a ramp will be incorporated in the design. The satellite workshops are provided to perform daily inspections, and minor services on all drill rigs, and LHDs. Weekly inspections are catered for boom-operated machines, due to the inherent slow tramming ability of these machines.

Surface Heavy Vehicle Workshop

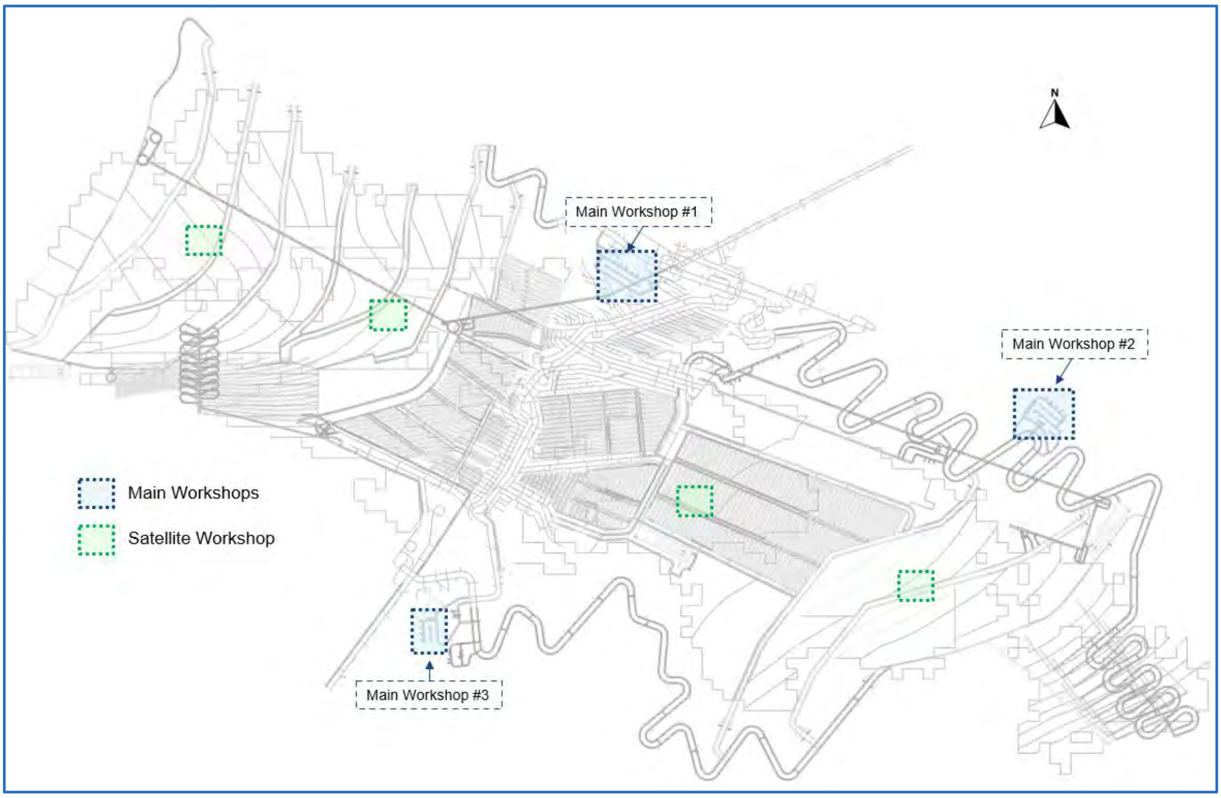
There are two surface workshops, a LDV workshop, and one heavy vehicle workshop. The surface workshops will be located in close proximity to the decline box-cut. Separate equipment wash bays will be located close to the workshops to ensure vehicles that report to the workshop are cleaned prior to entering for maintenance. The surface heavy vehicle workshop will be equipped to cater for all maintenance requirements on all surface, and underground production vehicles, as well as large mining support equipment. The layout of the surface heavy vehicle workshop is illustrated in Figure 16.50.

Figure 16.50 Service HV Workshop



Kakula will have three main workshop and four satellite workshops spread out to cover the footprint of the mining area. Figure 16.51 below indicates the approximate positions for the main, and satellite workshops.

Figure 16.51 Kakula Mine Workshops



DRA, 2022.

16.3.8.7 Fuel and Lubricant Distribution

Eight additional 3.2 MW generators are planned for Kakula which would need 8 x 70 m³ diesel tanks and 2 x 4.5 m³ oil storage tanks. Existing storage tanks and equipment at Kakula can be used to supply the new eastern and western batch tanks. New equipment required for Kakula eastern and western areas include 1 x 4.5 m³ batch tanks for diesel delivery underground, and 3 x 2.5 m³ batch tanks for oil delivery respectively for the eastern and western areas. Underground storage tanks are 3 x 15 m³ diesel tanks and 3 x 4.5 m³ oil tanks with 1 x m³ used oil tank respectively for the eastern and western areas. A total of thirty-two tanks are required for the main Kakula complex eastern and western sides.

16.3.8.8 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.

16.3.8.9 Electrical Power Requirements

The electrical power maximum demand for all additional areas at Kakula are indicated in Figure 16.52 and Table 16.28 below. 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 30-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 / production schedules were also applied to the running loads to obtain an MD load profile in kW.

Figure 16.52 Kakula Mine Life-of-Maximum Demand Profile

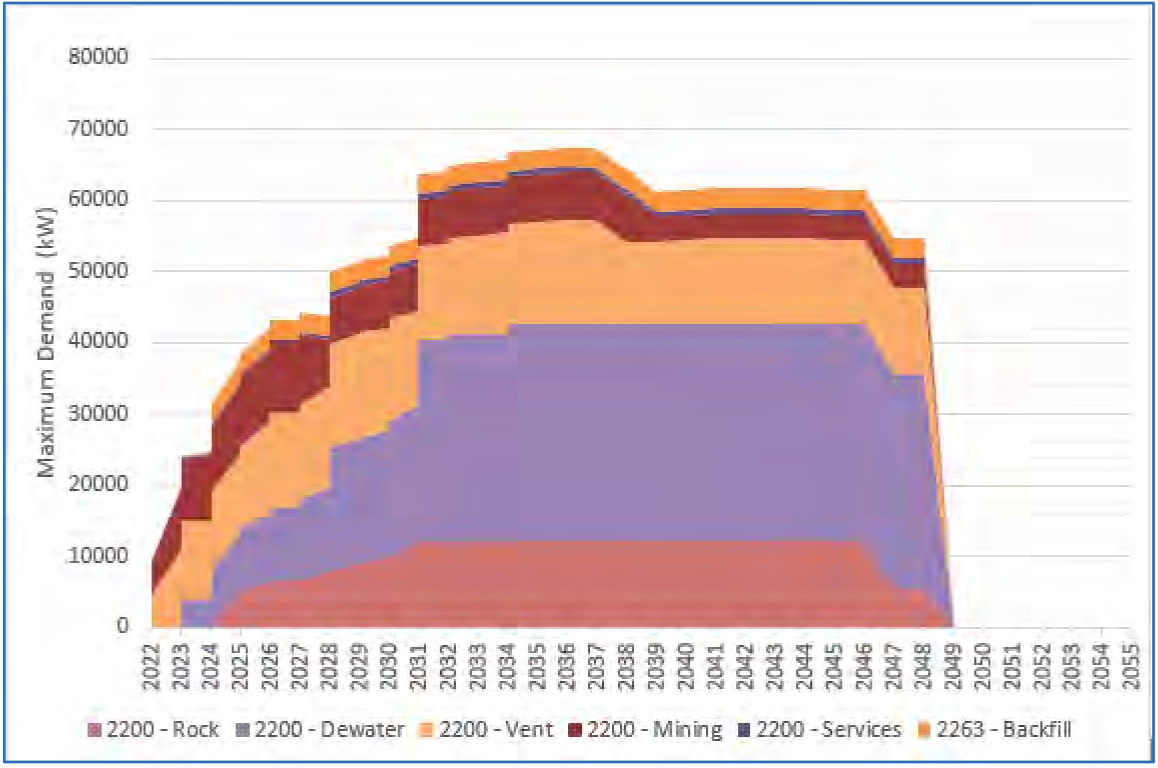


Table 16.28 Kakula Mine Life-of-Maximum Demand Profile

WBS – Area	Maximum Demand (kW) (2037)
2200 - Rock Handling	12,239
2200 - Dewatering	30,230
2200 - Ventilation and Cooling	14,801
2200 - Mining	6,600
2200 - Services	806
2263 - Backfill	2,800
Total	67,476

16.3.8.10 Power Distribution

The power distribution for the Kakula mine will form part of the existing infrastructure, with new overhead line and cable installed where required. The additional 33 kV distribution substations and 11 kV distribution substations have been allowed for at Kakula are listed below. Power is distributed from these substations to various equipment substations via dual supplies.

- KKM – 33 kV Vent Fan SE01
- KKM – 33 kV Vent Fan SW02
- KKM – 33 kV Vent Fan SW03
- KKM – 33 kV Fridge Plant #1
- KKM – 33 kV Fridge Plant #2
- KKM – 33 kV Fridge Plant #3
- KKM 11 kV NPD East Substation 2273-SUB-20
- KKM 11 kV NPD East Substation 2273-SUB-21
- KKM 11 kV NPD East Substation 2273-SUB-22
- KKM 11 kV NPD East Substation 2273-SUB-23
- KKM 11 kV SPD East Substation 2273-SUB-25
- KKM 11 kV SPD East Substation 2273-SUB-26
- KKM 11 kV SPD West Substation 2273-SUB-30
- KKM 11 kV SPD West Substation 2273-SUB-31
- KKM 11 kV NPD West Substation 2273-SUB-35
- KKM 11 kV NPD West Substation 2273-SUB-36
- KKM 11 kV NPD West Substation 2273-SUB-37
- KKM 11 kV NPD West Substation 2273-SUB-38

- KKM 11 kV NPD West Substation 2273-SUB-39

16.3.9 Equipment

All equipment is sized for a 9.2 Mtpa case to support drift-and-fill mining method. 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 based on the existing fleet and cover the major components for the operation.

16.3.9.1 Mobile Equipment

The mobile equipment is diesel-powered, rubber-tyred. Typical development equipment such as jumbo drills are 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.

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.

As Kakula is an operating mine with an existing mobile equipment fleet, some adjustments and assumptions were applied to the equipment calculation based on operational experience. The mobile equipment for Kakula is listed in Table 16.29.

Table 16.29 Kakula Mine Mobile Equipment List

Description	Maximum Number Required	Number of Units to Purchase	Number of Rebuilds
Drill Rig 282	34	46	42
LHD – 21 t	33	52	47
Haul Truck – 63 t	31	39	36
Concrete / Shotcrete Mixer Truck	15	11	N/A
Shotcrete Sprayer	15	11	N/A
Scissor Lift	23	50	N/A
Charmec – Explosives Loading Truck	15	43	N/A
Mining Support Equipment			
Explosives Transport Truck	15	47	N/A
Agicar – Concrete Mixing Truck – 12 m^3	6	5	N/A
Shotcrete – Backfill	6	12	N/A
Scissor Lift – Backfill	15	33	N/A
LVs	84	189	N/A
Personnel Carriers	20	46	N/A
Grader	5	15	N/A
Utility Equipment – Material	14	43	N/A
Utility Equipment – Maintenance	11	34	N/A
Telehandlers	15	47	N/A
Underground Mobile Crane	5	14	N/A
Skidsteer	11	6	N/A

16.3.9.2 Fixed Equipment

Table 16.30 lists the main fixed equipment types that will support the mining operation at full production. A detailed mechanical equipment list was developed during the Kamoa-Kakula 2023 PFS.

Table 16.30 Kakula Mine Fixed Equipment

Services	Description
Materials Handling	Truck Tips
	Static Hydraulic Rock Breakers
	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 Pumps
Electrical and Communications	Main Substation
	Motor Control and Mine Power Centres
	VSD's
Safety and Miscellaneous	Air Compressors (Surface compressors required for refuge chambers)
	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
	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

16.4 Kakula West Underground Mining

16.4.1 Introduction

Based on updated design criteria, the mining method, mine design, and production schedule have been updated from previous studies. Mining method 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 selected and the resultant mine designs and schedules.

The mining method used for the Kakula West deposit is drift-and-fill. The Kakula West Mineral Reserve by mining method is summarised in Table 16.31.

Table 16.31 Kakula West Probable Mineral Reserves by Mining Method

Production by Mining Method	Ore (Mt)	TCu (%)	Fe (%)	As (%)	S (%)
Ore Development	0.85	3.17	4.55	0.00	1.04
Drift-and-Fill	88.73	3.88	4.88	0.00	1.27
Total Ore*	89.58	3.87	4.88	0.00	1.27

*May not sum to total due to rounding.

16.4.2 Mining Design Parameters

16.4.2.1 Ore and Waste Properties

The Kakula West 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/t NSR (net smelter return). The ore zone density has been defined as using a greater than 2.4% Cu (total copper grade) cut-off. The swell factor for development is 50%.

Table 16.32 details the bulk density parameters of the ore and surrounding waste rock of the Kakula West deposit.

Table 16.32 Bulk Density/In Situ by Area

Bulk Density/In Situ	Min (t/m^3)	Max (t/m^3)	Average (t/m^3)
Ore	2.27	3.23	2.81
Hanging wall	2.39	3.18	2.80
Footwall	2.21	3.04	2.67

16.4.3 Mine Planning

16.4.3.1 Lateral Development

The southern declines have a maximum gradient of ±9°. The conveyor drift is 7.5 m W x 6.0 m H with 1.5 m arch corners, and 6 m W x 6.0 m H with 1.5 m arch corners for the service drift. 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 6 m W x 6 m H, with 1.5 m arch corners unless otherwise specified.

Conveyor drifts have a maximum gradient of ±9°. They are 7.5 m W x 6 m H (1.5 m arch corners), with re-mucks located every 300 m.

All perimeter drifts are 6 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 crosscuts 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° and 1.5 m arch corners.

16.4.3.2 Vertical Development

Vertical development consists of ventilation raises, bins, and boreholes. All ventilation raises are excavated with a raisebore drill. All ventilation shafts are designed at 6 m in diameter. Access drifts to the ventilation shafts are 6 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.

16.4.3.3 Drift-and-Fill

The primary mining method for the Kamoa-Kakula deposits (drift-and-fill) was selected based on the results of a trade-off study which compared six different variations of drift-and-fill mining to identify the most suitable. The selected drift-and-fill mining method is explained in detail in Section 16.2.2.

A typical mining block with production drifts perpendicular to the connection drift is 198 m wide. Each mining block consist of three mining units. Each mining unit comprises three primary headings, three secondary headings, and three tertiary headings.

The production drift cross-sectional shape differs with the type of production heading. Typical primary and secondary production drifts have an 8 m wide arch (1.5 m arch corners), with a maximum height up to 7.6 m. Slipe drifts are a 6 m wide arch (1.5 m arch corners), with a maximum height up to 7.6 m.

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 (9°).

16.4.4 Backfill

Paste fill will be the primary backfilling strategy for the Kakula West 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.

16.4.5 Mine Access Designs

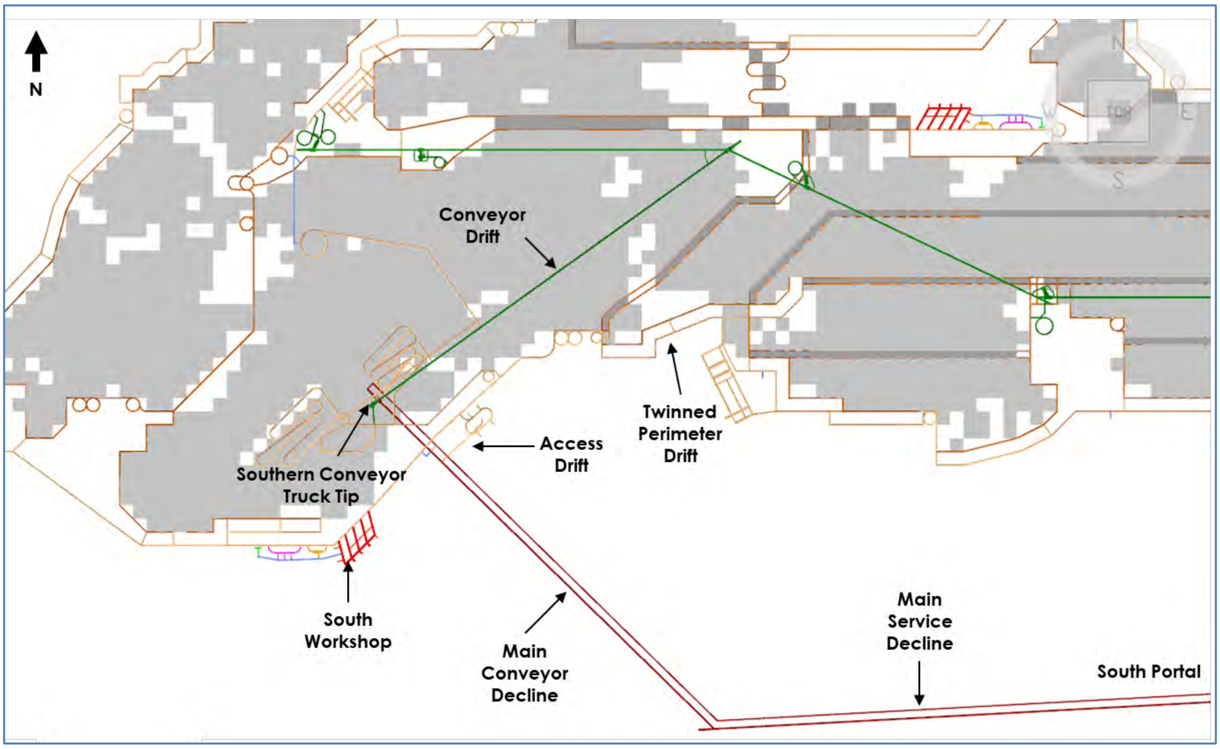
Main Declines

The deposit is accessed via a twinned decline on the southern side. There is one box-cut developed for access to the underground workings. The southern box-cut incorporates two portals.

One of the south declines is the primary mine service access, and the other decline is a conveyor haulage drift. The service decline has dimensions of 6.0 m W x 6.0 m H, with the conveyor decline 7.5 m W x 6.0 m H. Both southern declines have a maximum gradient of ±9° gradient.

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 5,205 m from the portal, 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 shown in Figure 16.53.

Figure 16.53 Kakula West Mine Main Access Development



OreWin, 2023.

Perimeter Service and Conveyor Drifts

From the bottom of the south decline, 6 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 conveyor drifts are located central to the orebody and run along the east–west axis. The western conveyor system extends to the lower north-western extremity of the orebody and converges onto the central conveyor system via a silo arrangement. The eastern system extends to the lower south-eastern limit and converges onto the central conveyor system. The central conveyor system then transports ore to surface via the southern main conveyor decline. Conveyor drifts are 7.5 m W x 6.0 m H and are driven at a maximum gradient of 9.0°. Section 16.4.8.4 covers the rock handling systems in greater detail.

Mining Areas

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.

16.4.6 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 6.2 Mtpa ore production. Based on a 360-day operating schedule, the production goal is to sustain full production for 10-years. The upper areas in the western side of Kakula West are more steeply-dipping and have more complex ground conditions than the deeper areas to the north, and east, and Kakula West. Ground support recommendations have recognised this, and development and production scheduling have taken it in to account by allowing lower productivities is the more complex areas.

Table 16.33 summarises the LOM development and production results.

The following conditions were used in developing the LOM schedule:

- Proximity to the main accesses and initial 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.

Table 16.33 Kakula West Mine Life-of-Mine Development and Production Summary

Waste Development	
Lateral (m)	119,091
Lateral (kt)	12,016
Vertical (m)	11,496
Vertical (kt)	731
Production by Mining Method	
Ore Development (m)	9,011
Ore Development (kt)	854
Drift-and-Fill (kt)	88,727
Total Ore Production	
Total Ore Development (kt)	854
Total Production (kt)	88,727
Total Tonnes (kt)	89,582
Diluted Grade	
TCu (%)	3.87
S (%)	1.27
As (%)	0.0001
Fe (%)	4.88

1. Notes: Vertical development includes boreholes.
2. Stope shapes designed on an NSR cut-off value of US$100/t NSR.

16.4.6.1 Production Rates

Effective Operating Hours

The effective operating hours per shift are summarised in Table 16.34 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 16.34 Kakula West Mine Shift Rotations and Effective Operating Hours Calculations

Shift Cycle	Calculations
Days per Year	360 days
Number of Crews in Rotation	3
Shifts per Day	2
Shift Duration	12 h
Travelling Time – In	19.50 min / 0.32 h
Travelling 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, and external consultants) and compared with inputs. The cycles were updated accordingly following team discussions. Mine productivities and schedule are based on a development rate of 120 m/month for all primary development.

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

The production rate in ore tonnes per month is highly variable as it is an outcome of the estimated development advance rate, combined with ore thickness, dilution parameters, and ore density.

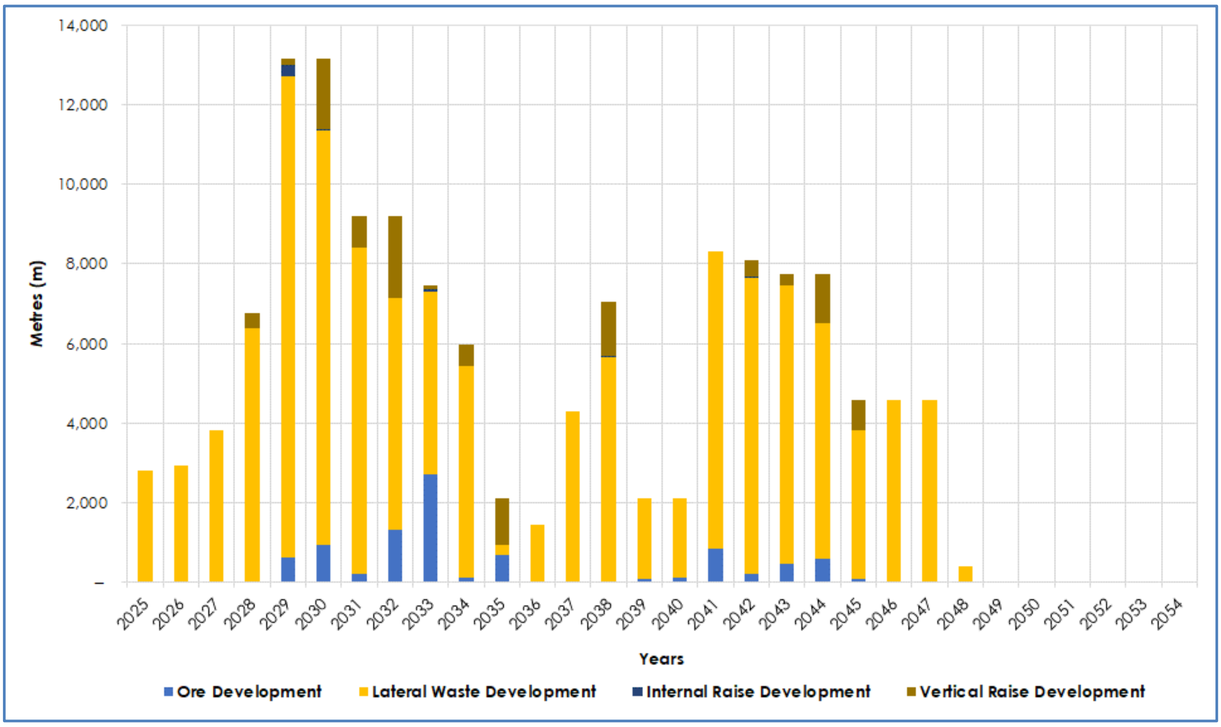
To determine the drift-and-fill production rates, development rates for primary, secondary, and tertiary drifts were combined with paste filling, cable bolting, and end-of-shift blasting restrictions in a block configuration to determine the net block production rate for use in the schedule. The net block production rate was then applied to the drift-and-fill mining shapes within the schedule. The production rate was adjusted depending on the height of the production heading in order to better represent the change in condition. Paste fill barricade construction/placement as well as installation of split sets (geotechnical supports) was not considered in the cycle calculations as it will be completed off critical task.

16.4.6.2 Development Schedules

Development schedule

The life-of-mine development schedule targets the areas required to support the LOM plan. This includes excavating the declines, perimeter service drifts, conveyor drifts, and key infrastructure associated with truck tips, ventilation, dewatering, and maintenance facilities in advance of production areas. For Kakula West, there is a four year pre-production period (2025–2028) which consists of setup of key infrastructure to enable ore production.
Figure 16.54 illustrates the development metres associated with the LOM activities.

Figure 16.54 Kakula West Mine Life-of-Mine Development Schedule



OreWin, 2023.

16.4.7 Mine Production Plan and Scheduling

The Kakula LOM milling schedule targets 9.2 Mtpa processed. As Kakula West begins to ramp-up, Kakula production will ramp down, with the total ore provided to the mill remaining constant at 9.2 Mtpa. The initial ramp-up scheduled tonnage for Kakula West will be supplemented with ore from Kakula in order to meet the processing target. Initially, Kakula West will ramp-up and remain at a steady state of 3.2 Mtpa for nine-years (2029–2037). Table 16.35 presents the annual ramp-up scheduled tonnes for Kakula West.

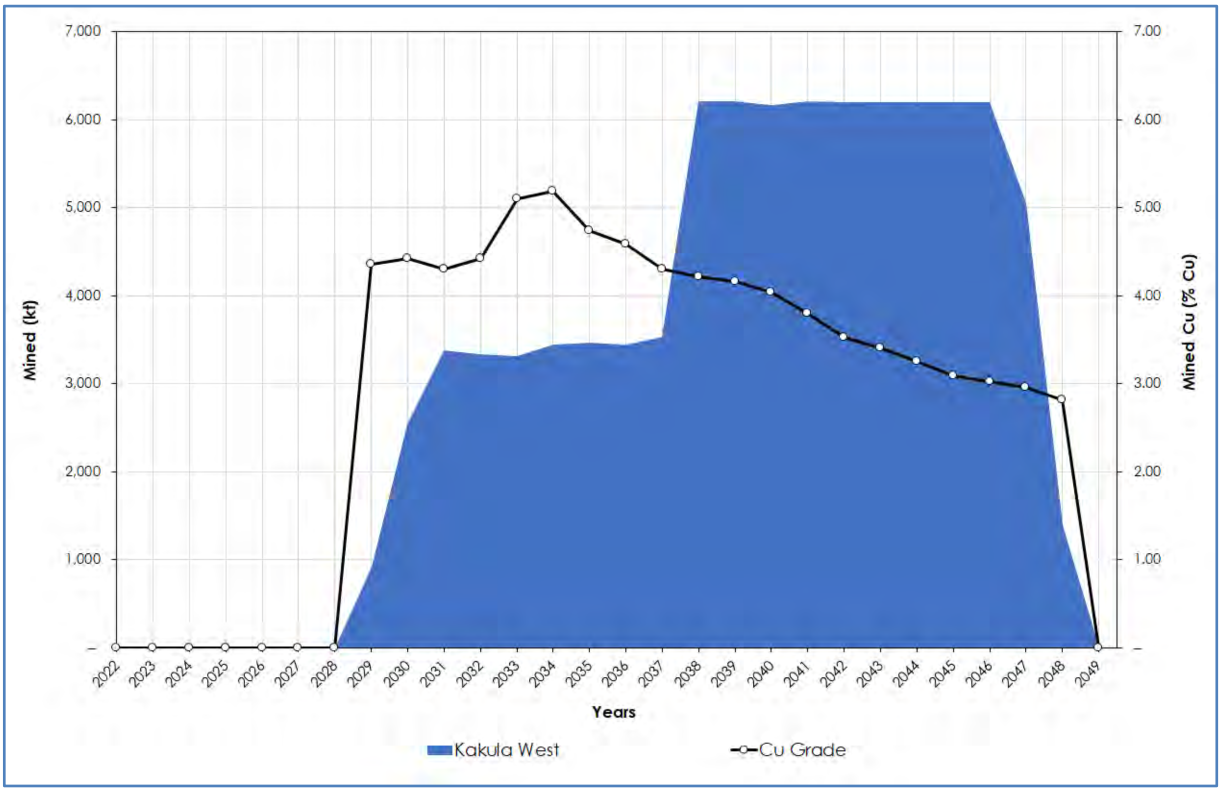
Table 16.35 Kakula West Mine Ramp-up Scheduled Tonnage

Production Schedule	Years	Scheduled Tonnes (kt)
Ramp-Up (2029)	1	924
Ramp-Up (2030)	1	2,532
Ramp-Up Total	2	3,455

Life-of-Mine Production Schedule

Kakula West will ramp-up and remain at a steady state of 3.2 Mtpa for nine-years (2029–2037). As Kakula West begins production (3.2 Mtpa), Kakula production will ramp-down to 6.0 Mtpa so that the total amount of material supplied to the Kakula mill remains at 9.2 Mtpa. Kakula West will then ramp-up and maintain its full production rate (6.2 Mtpa) for nine-years (2038–2046) before beginning to taper off as the deposit is depleted (2047–2048). As Kakula West ramps-up to full production (6.2 Mtpa), Kakula production will ramp down to 3.0 Mtpa until LOM. The relationship between Kakula and Kakula West is continued for the Kakula West LOM so that both deposits ramp down to finish production together. The mining blocks are scheduled so that a higher TCu value is achieved early in the mine life. Figure 16.55 illustrates the LOM production schedule and copper grade.

Figure 16.55 Kakula West Mine Life-of-Mine Production Schedule and Copper Grade



OreWin, 2023.

16.4.8 Underground Infrastructure

16.4.8.1 Mine Ventilation System - Kakula West

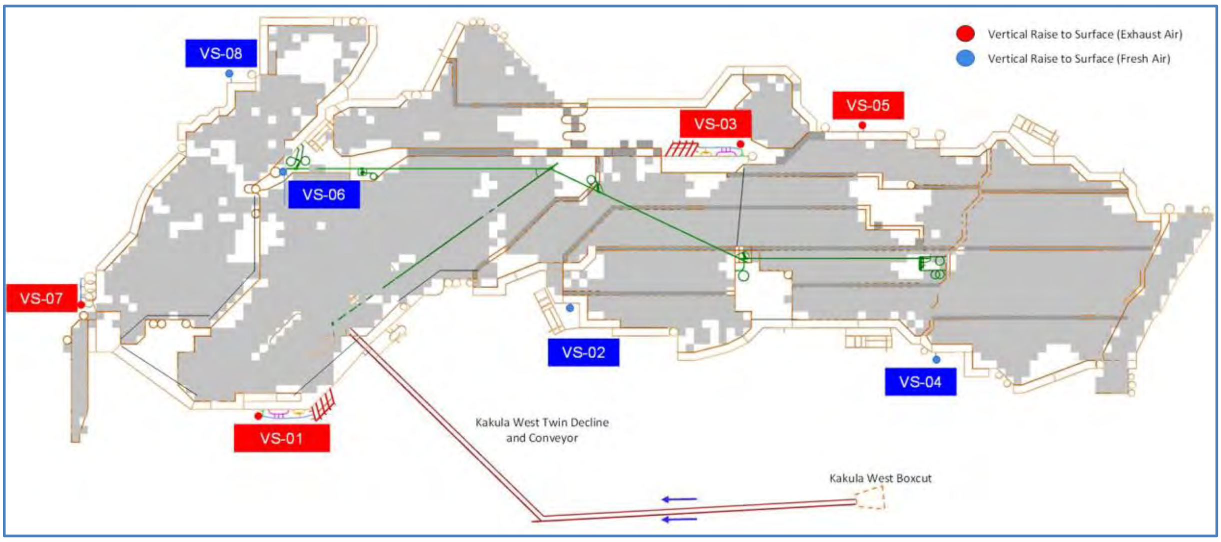
The underground mobile equipment fleet is diesel powered, and mine air cooling is required to maintain underground working air quality within the appropriate limits.

The following assumptions were considered in the ventilation design to maintain safe operating conditions underground and to abide by applicable legislative requirements. Australian and 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.
- Primary and secondary leakage rates used for preliminary airflow estimates for when the mine is fully developed are 10% and 20% respectively. These factors are used 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 30 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.
- Heat load factors for diesel equipment have been split into four types. The following types of trucks, loaders, auxiliary and supporting equipment have respective heat load factors of 1.1, 1.0, 0.8, and 0.3, applied as linear activity tracks in the Ventsim modelling.
- Diesel engines are assumed to have a power conversion efficiency of 35%.

The mine layout schematic illustrating the location of the ventilation shafts is shown in Figure 16.56. The ventilation system is designed to provide fresh air through four central shafts. The ventilation air will naturally flow from the intake ventilation shafts located central to the orebody and then through the perimeter drives to the extremities of the orebody. Air will return through return airways located at the extremities of the orebody. VS-01 will ventilate the conveyor belt directly to return. 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 16.56 Kakula West Mine Layout with Ventilation Shaft Locations



OreWin, 2023.

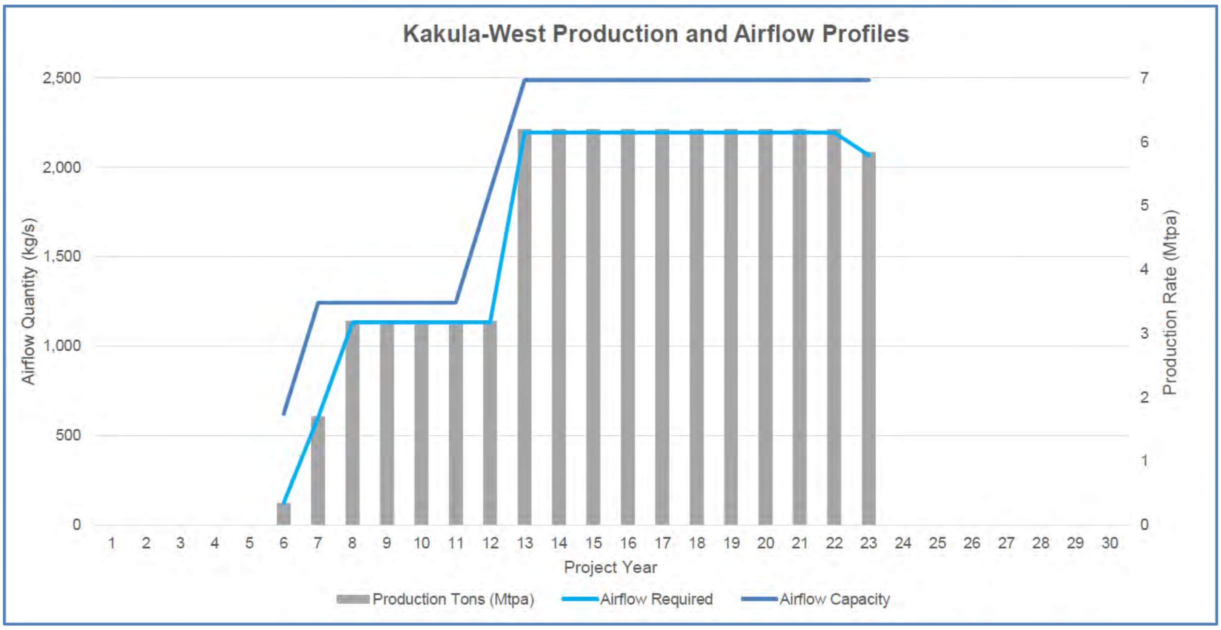
A summary of the primary ventilation fans is provided in Table 16.36. The LOM airflow requirements are shown in Figure 16.57.

Table 16.36 Kakula West Mine Primary Main Ventilation Fan Requirements

Raise Location	No. of Fans (in parallel)	Operating Range (kg/s)	Peak Airflow (kg/s)	Peak Total Pressure at Collar (kPa)	Estimated Power (kW)
VS-01	3	660 - 750	560	1.5 – 1.8	1,455 - 1,930
VS-03	3	660 - 750	560	1.5 – 2.2	1,455 – 2,360
VS-05	3	660 - 750	560	1.5 – 2.2	1,455 – 2,360
VS-07	3	660 - 750	560	1.5 – 2.2	1,455 – 2,360

VS-01, VS-03, VS-05, and VS-07 fan stations will be required in PY6, PY7, PY12 and PY13 respectively. Each fan station is equipped with three fans in a trifurcated fan arrangement.

Figure 16.57 Kakula West Mine Life-of-Mine Airflow Requirements



OreWin, 2023.

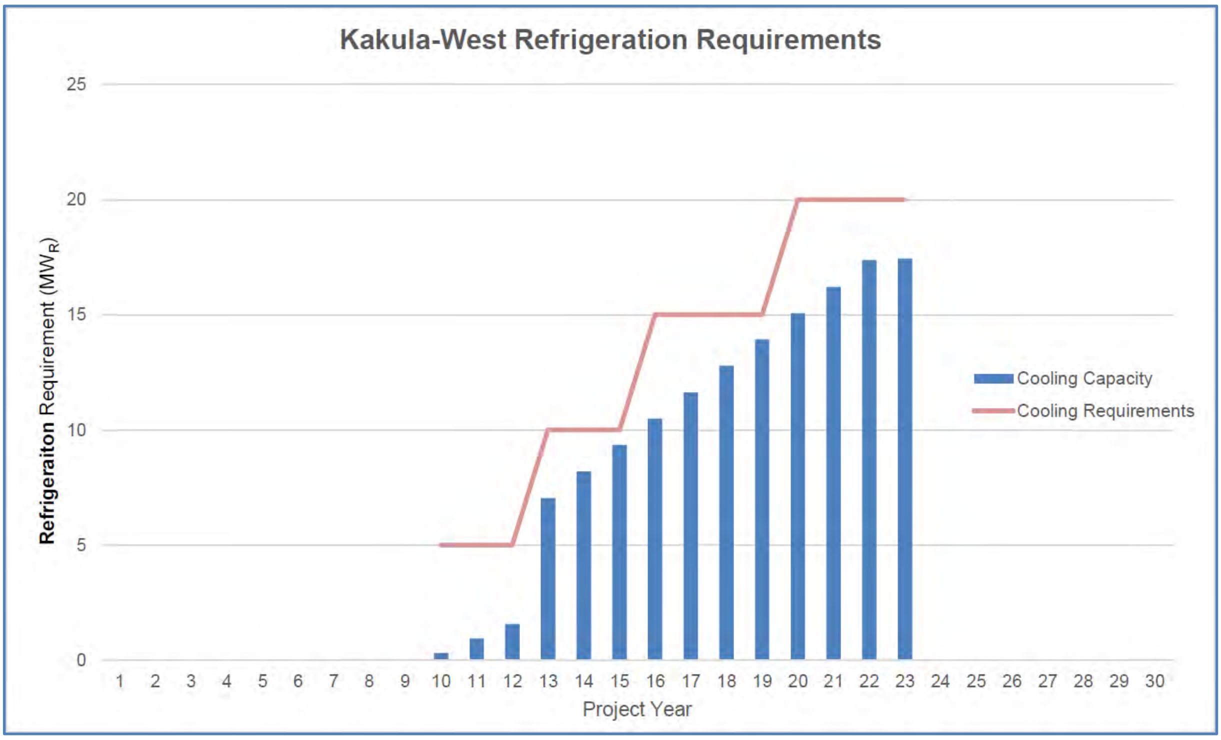
The model shows the primary ventilation requirements of 2,230 kg/s at the peak production rate of 6.2 Mtpa and a maximum depth of 930 m.

16.4.8.2 Mine Air Cooling Facilities – Kakula West

The heat load and refrigeration requirements of the mine are shown in Figure 16.58. The cooling requirements were estimated from first principles and confirmed by VentSim models which considered auto-compression, surrounding and broken rock heat, vehicle heat, and all other heat components. The model indicated that 18 MW of cooling is required at the peak production rate of 6.2 Mtpa while producing up to 990 m below surface.

The refrigeration machines will be located in a refrigeration plant room near Kakula West, serving four modular type BACs at the top of the new intake ventilation shafts (VS-02, VS-04, VS-06, and VS-08). To provide a total cooling duty of 18 MWR (megawatts of refrigeration) to underground, 20 MWR of refrigeration machine duty will be required (including cooling losses). Four mechanical refrigeration modules will be phased-in as shown in Figure 16.58. The refrigeration machines will be phased-in at 5 MWR modules and 4.5 MWR BACs.

Figure 16.58 Kakula West Mine Life-of-Mine Refrigeration Load



OreWin, 2023.

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 16.37.

Table 16.37 Kakula West Mine Auxiliary Ventilation Fan Requirements

Location	Flow per Fan (m^3/s)	Fan Total Pressure (Pa)	Fan Size diameter (m)	Duct Type	Estimated Power (kW)
Development Headings	36	2,300	1.40	Flexible	110
Drift-and-fill Headings	36	2,300	1.40	Rigid and Flexible	110

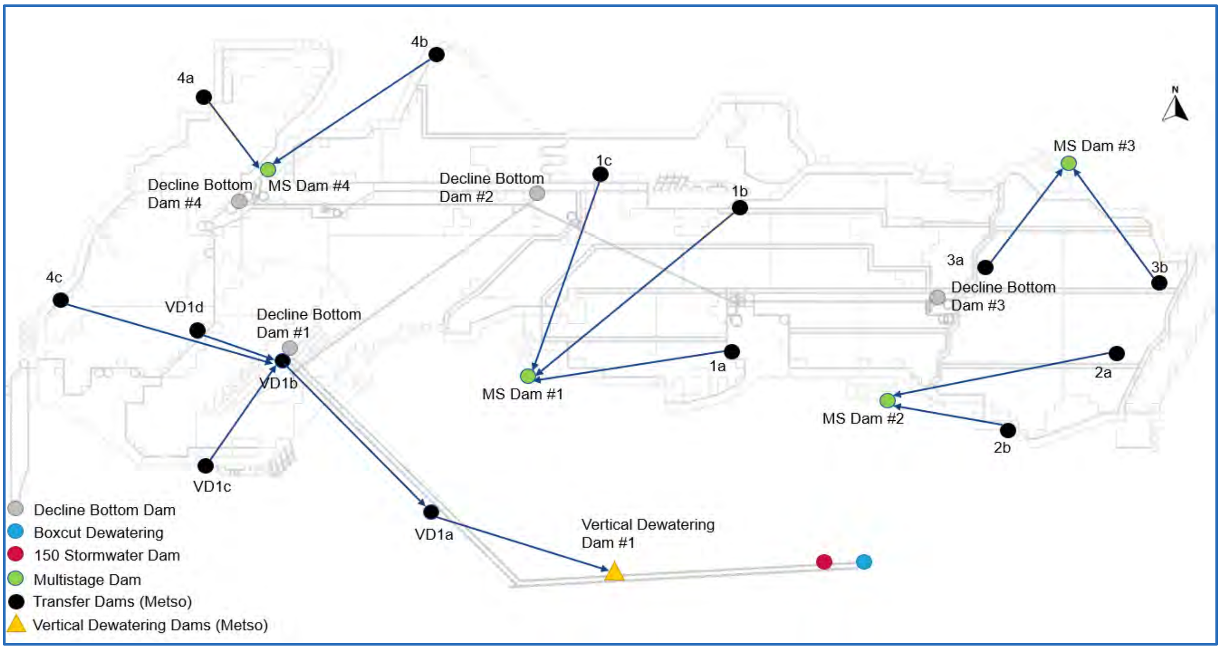
16.4.8.3 Mine Dewatering

Kakula West will have a main decline pumping system which consists of a shaft bottom transfer dam pumping up to a series of intermediate dams. The final intermediate dam is a vertical dewatering dam which will pump out to surface via a dewatering borehole. The main decline pumping arrangement is capable of handling 200 – 1,000 l/s.

In addition to the main decline pumping system, four additional multistage (MS Dams) dewatering pump stations will be located throughout the Kakula West orebody – indicated in Figure 16.59. Each multistage dewatering pump station will be capable of pumping 1,500 l/s to surface through dewatering boreholes. Kakula West is expected to reach a maximum of approximately 4,000 l/s fissure water inflow.

Each multistage dewatering pump station will be fed by two or three transfer dam pump stations. The transfer dam pump stations, indicated as black dots in Figure 16.59, will have the capacity to pump 200 – 1,000 l/s to the main multistage dewatering dams. The transfer dams will be strategically positioned to cover the required footprint of the mined ore body.

Figure 16.59 Kakula West Mine Dewatering Layout

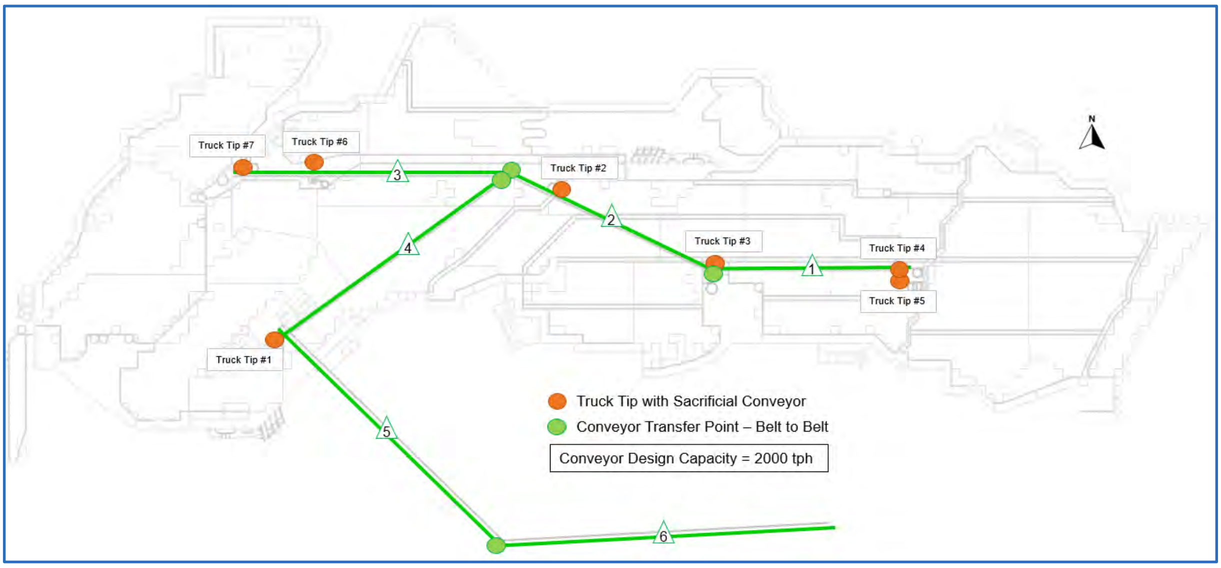


DRA, 2022.

16.4.8.4 Rock Handling

The Kakula West underground rock handling system comprises a Main Decline conveyor fed by a North–South conveyor from the north. The East Feed, and West Feed conveyors feed into the North–South conveyor, extending the rock handling network to the east and west of the ore deposit. Figure 16.60 illustrates the routing of the underground conveyors in Kakula West.

Figure 16.60 Kakula West Mine Underground Rock Handling Conveyor Routing



DRA, 2022.

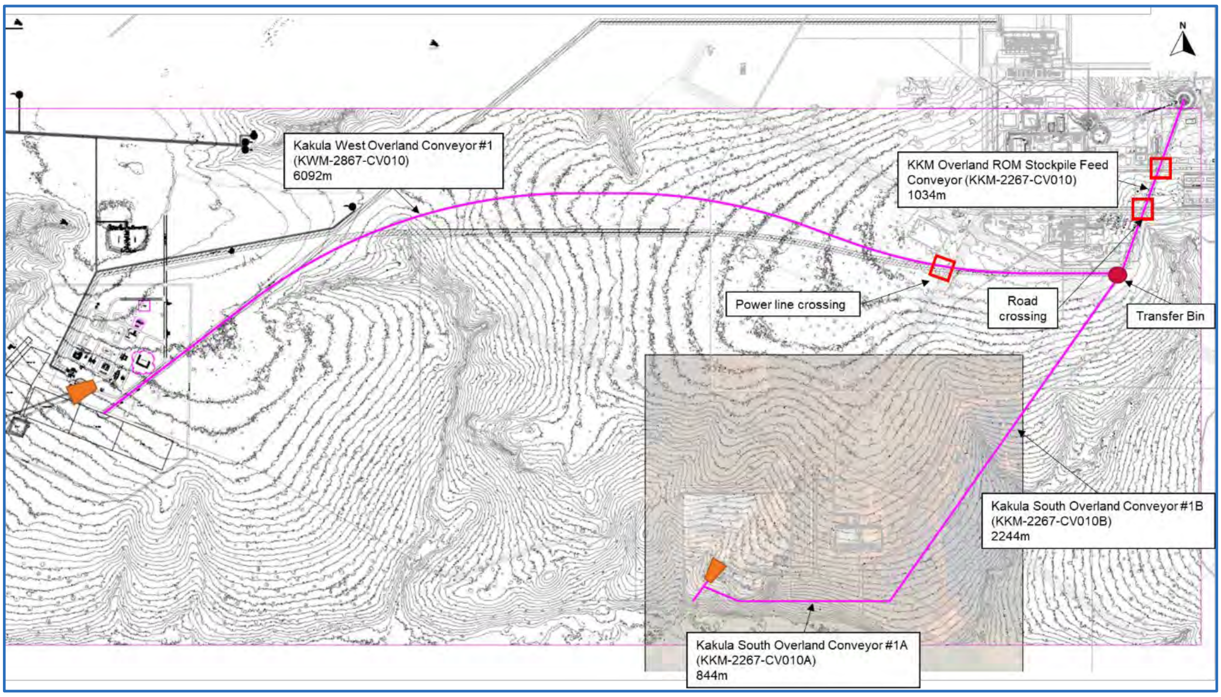
The rock handling configuration for Kakula West main decline was based on the outcomes of a trade-off study, which compared seven different configurations, and a two-conveyor belt system, which was used in the PFS, with a design capacity of 2,000 tph.

The main decline conveyor feeds the surface ROM transfer conveyor that is installed with a diverter chute, to split the flow to either the bypass stockpile, or to the overland conveyor crusher system. In the case where the crusher, or the overland conveyor system is unavailable, the bypass system can be utilised to ensure there is no production stoppages.

Material, diverted onto the bypass system, will report onto a bypass stockpile feed conveyor, and dumped on a bypass stockpile. A loader and truck will in turn will be used to load, haul, and tip the material onto the bulk stockpile dump. Reclaimed ore will be transported to a bulk reclaim tip, which will discharge material at a controlled feed rate onto the ROM transfer conveyor. Provisions at the bulk reclaim tip is made for dust suppression, oversize rock breakage, and tramp iron removal.

All material, either reporting directly from the main decline conveyor or from the reclaim system, will enter a crusher before being fed onto the overland conveyor. The overland conveyor runs back as a single conveyor up to where it ties-in with the Kakula South surface conveyor at a transfer point near the Kakula Backfill plant. The Kakula West overland conveyor in relation to the Kakula South overland conveyor is illustrated in Figure 16.61.

Figure 16.61 Kakula and Kakula West overland conveyors



DRA, 2022.

16.4.8.5 Materials Handling Logistics

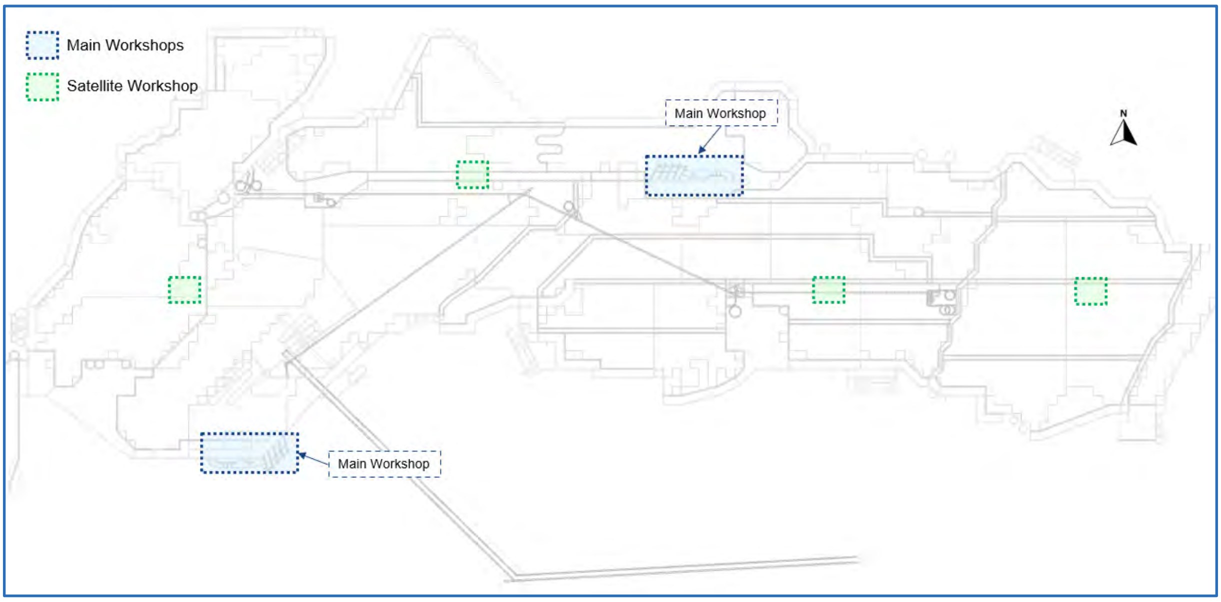
The material handling procedures for Kakula West will be the same as those for Kakula. Men, and materials will be transported underground via the service decline, while emulsion, shotcrete, fuel, and lube will be transferred underground using boreholes. These facilities will be established in the vicinity of the underground workshops.

16.4.8.6 Workshops

Kakula West was based on the same workshop designs and philosophies as described in 16.3.8.6 (Under Kakula Workshops) and will have the same number surface workshops.

Underground, two centrally located main workshops and four satellite workshops will be constructed to cover the mining footprint. Figure 16.62 indicates the approximate positions for the main and satellite workshops.

Figure 16.62 Kakula West Mine Workshops



DRA, 2022.

16.4.8.7 Fuel and Lubricant Distribution

Kakula West is a new mine and will require new infrastructure. The two surface workshops each will require one 70 m³ diesel tank, and four oil storage tanks. Two underground workshops are planned, and each will require the normal storage of three 15 m³ diesel storage tanks and four oil storage tanks. One 4.5 m³ batch tank and three 2.5 m³ batch tanks will be supplied for the supply of hydrocarbons to the underground workings.

The total number of tanks planned for Kakula West are thirty-three.

16.4.8.8 Toilet System

As described under Kakula mine.

16.4.8.9 Electrical Power Required

The electrical power requirements for all areas at Kakula West are indicated in Figure 16.63 and Table 16.38 below.

Figure 16.63 Kakula West Mine Life of Maximum Demand Profile

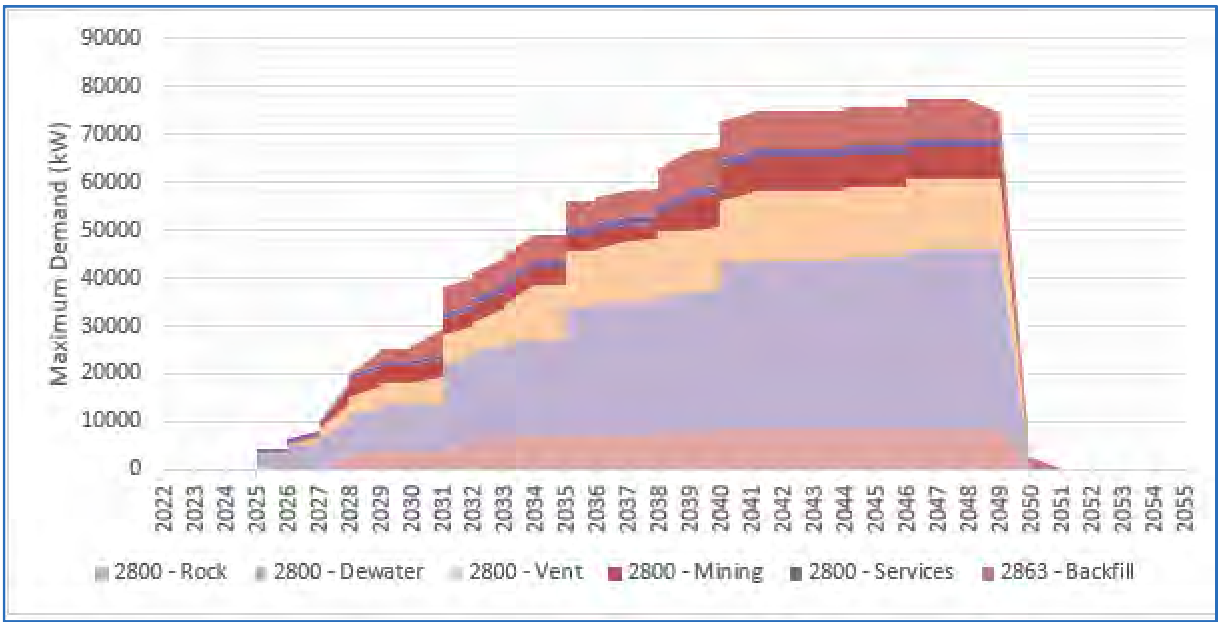


Table 16.38 Water Handling Power Requirements

WBS – Area	Maximum Demand (kW) (2046)
2800 – Rock	8,466
2800 – Dewater	37,590
2800 – Vent	14,401
2800 – Mining	7,000
2800 – Services	1,720
2863 – Backfill	8,000
Total	77,177

16.4.8.10 Power Distribution

A new dedicated 33 kV overhead powerline is to be installed from the 33 kV Kakula KCS substation to feed the Kakula West 33 kV Portal substation.

The Kakula West Portal substations will be equipped with MV switchgear, and 33/11 kV transformers.

Underground power distribution will be via 11 kV cable distribution substations. For smaller loads, the mini substations will provide suitable LV supplies (690 V, 400 V, 1000 V), as required in the area.

The 33 kV distribution substations and 11 kV distribution substations allowed for are listed below. Power is distributed from these substations to various equipment substations.

- KWM 33 kV Portal Substation
- KWM 11 kV Portal Substation
- KWM Vent Fan #1 Substation
- KWM – 33 kV VS-01 Substation
- KWM – 33 kV VS-03 Substation
- KWM – 33 kV VS-05 Substation
- KWM – 33 kV VS-07 Substation
- KWM – 33 kV Refrigeration Plant #1 Substation
- KWM – 33 kV Refrigeration Plant #2 Substation
- KWM – 33 kV Refrigeration Plant #3 Substation
- KWM – 33 kV Refrigeration Plant #4 Substation
- KWM – 33 kV Backfill Plant Substation
- KWM – 11 kV Substation #1, 2873-SUB-001
- KWM – 11 kV Substation #2, 2873-SUB-002
- KWM – 11 kV Substation #3, 2873-SUB-003
- KWM – 11 kV Substation #4, 2873-SUB-004
- KWM – 11 kV Substation #5, 2873-SUB-005
- KWM – 11 kV Substation #6, 2873-SUB-006
- KWM – 11 kV Substation #7, 2873-SUB-007
- KWM – 11 kV Substation #8, 2873-SUB-008
- KWM – 11 kV Substation #9, 2873-SUB-009
- KWM – 11 kV Substation #10, 2873-SUB-010

16.4.9 Equipment

All equipment is sized for a 6.2 Mtpa case to support a drift-and-fill mining method. 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 based on the existing fleet and cover the major components for the operation.

16.4.9.1 Mobile Equipment

The mobile equipment is diesel-powered, rubber-tyred. 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.

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 for Kakula West is listed in Table 16.39.

Table 16.39 Kakula West Mine Mobile Equipment List

Description	Maximum Number Required	Number of Units to Purchase	Number of Rebuilds
Drill Rig 282	23	27	23
LHD – 21 t	20	27	20
Haul Truck – 63 t	18	20	14
Concrete / Shotcrete Mixer Truck	11	11	N/A
Shotcrete Sprayer	11	11	N/A
Scissor Lift	16	30	N/A
Charmec – Explosives Loading Truck	11	27	N/A
Mine Support Equipment			
Explosives Transport Truck	11	29	N/A
Agicar – Concrete Mixing Truck – 12 m^3	4	4	N/A
Shotcrete – Backfill	4	7	N/A
Scissor Lift – Backfill	11	21	N/A
LVs	58	112	N/A
Personnel Carriers	14	27	N/A
Grader	3	9	N/A
Utility Equipment – Material	10	26	N/A
Utility Equipment – Maintenance	8	19	N/A
Telehandlers	11	29	N/A
Underground Mobile Crane	3	8	N/A
Skidsteer	8	8	N/A

16.4.9.2 Fixed Equipment

As described under Kakula, Section 16.3.9.2.

16.5 Kamoa 1 Underground Mining

16.5.1 Introduction

Based on updated design criteria, the mining method, mine design, and production schedule have been updated from previous studies. Mining method selection focussed 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 selected and the resultant mine designs and schedules.

The mining method used for the Kamoa 1 deposit is drift-and-fill. The Kamoa 1 Mineral Reserve by mining method is summarised in Table 16.40.

Table 16.40 Kamoa 1 Mine Probable Mineral Reserves by Mining Method

Production by Mining Method	Ore (Mt)	TCu (%)	Fe (%)	As (%)	S (%)
Ore Development	4.49	3.85	5.89	0.00	2.46
Drift-and-Fill	116.35	3.74	5.82	0.00	2.48
Total Ore*	120.84	3.74	5.83	0.00	2.48

*May not sum to total due to rounding.

16.5.2 Mine Design Parameters

16.5.2.1 Ore and Waste Properties

The Kamoa 1 deposit is a large stratiform copper deposit, typical of sediment hosted deposits. The deposit is tabular, with dips varying from 0–35° with an average dip of 17°. The thicknesses varies from 3 m to 6 m. The ore zone density has been defined as using a greater than 2.4% Cu (total copper grade) cut-off. The swell factor for development is 50%.

Table 16.41 details the bulk density parameters of the ore and surrounding waste rock of the Kamoa 1 deposit.

Table 16.41 Bulk Density/In Situ by Area

Bulk Density/In Situ	Min (t/m^3)	Max (t/m^3)	Average (t/m^3)
Ore	2.27	3.23	2.81
Hanging wall	2.39	3.18	2.80
Footwall	2.21	3.04	2.67

16.5.3 Mine Planning

16.5.3.1 Lateral Development

The western declines have a maximum gradient of ±9.5°. The conveyor drift is 7.5 m W x 6.0 m H with 1.5 m arch corners, and 6 m W x 6.0 m H with 1.5 m arch corners for the service drift. 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 6 m W x 6 m H, with 1.5 m arch corners unless otherwise specified.

Conveyor drifts have a maximum gradient of ±9°. They are 7.5 m W x 6 m H (1.5 m arch corners), with re-mucks located every 300 m.

All perimeter drifts are 6 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 crosscuts 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° and 1.5 m arch corners.

16.5.3.2 Vertical Development

Vertical development consists of ventilation raises, bins, and boreholes. All ventilation raises are excavated with a raisebore drill. All ventilation shafts are designed at 6 m in diameter. Access drifts to the ventilation shafts are 6 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.

16.5.3.3 Drift-and-Fill

The primary mining method for the Kamoa-Kakula deposits (drift-and-fill) was selected based on the results of a trade-off study which compared six different variations of drift-and-fill mining to identify the most suitable. The selected drift-and-fill mining method is explained in detail in Section 16.2.2.

A typical mining block with production drifts perpendicular to the connection drift is 198 m wide. Each mining block consist of three mining units. Each mining unit comprises three primary headings, three secondary headings, and three tertiary headings.

The production drift cross-sectional shape differs with the type of production heading. Typical primary and secondary production drifts have an 8 m wide arch (1.5 m arch corners), with a maximum height up to 7.6 m. Slipe drifts are a 6 m wide arch (1.5 m arch corners), with a maximum height up to 7.6 m.

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 (9°).

16.5.4 Backfill

Paste fill will be the primary backfilling strategy for the Kamoa 1 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 western perimeter declines. Distribution pipes installed in the connection drifts will then deliver the paste fill to the production areas.

16.5.5 Mine Access Designs

Box-Cuts

There is one box-cut developed for access to the underground workings. The western box-cut incorporates two portals.

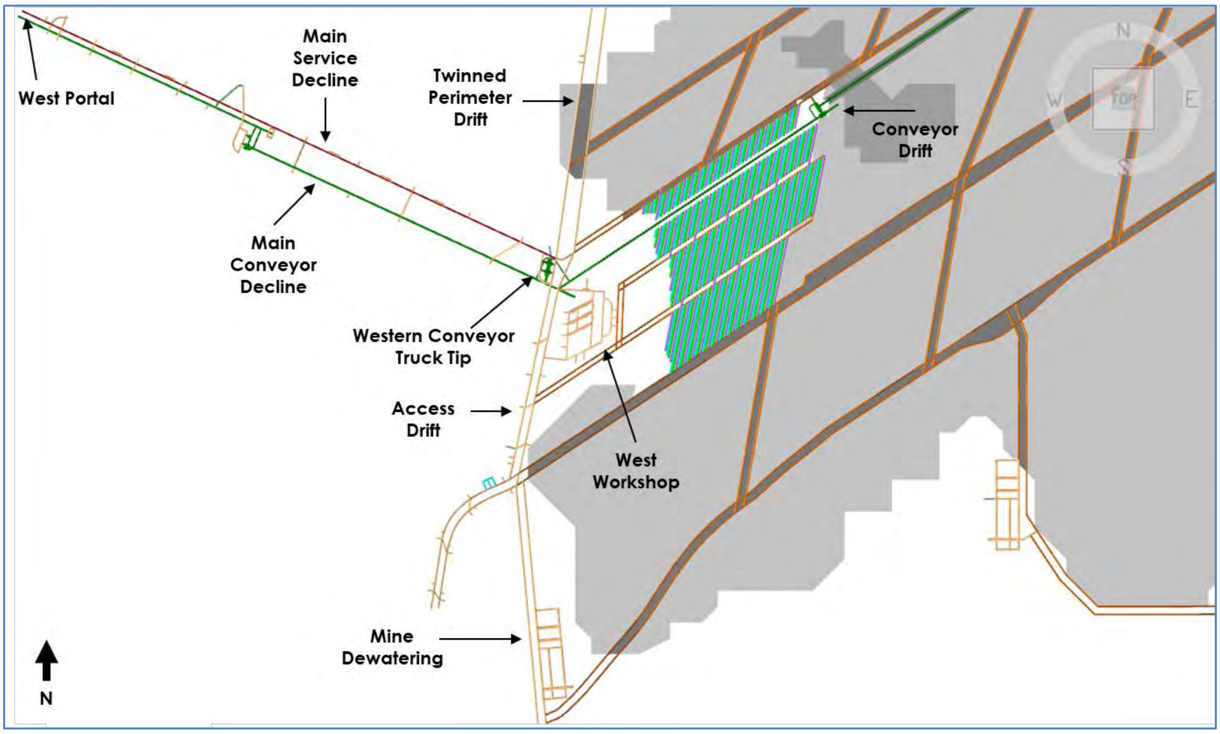
Main Declines

The deposit is accessed via a twinned decline on the western side.

One of the west declines is the primary mine service access, and the other decline is a conveyor haulage drift. The service decline has dimensions of 6.0 m W x 6.0 m H, with the conveyor decline 7.5 m W x 6.0 m H. Both western declines have a maximum gradient of ±9.5° gradient.

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 1,875 m from the portal, 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 shown in Figure 16.64.

Figure 16.64 Kamoa 1 Mine Main Access Development



OreWin, 2023.

Perimeter Service and Conveyor Drifts

From the bottom of the west decline, 6 m w x 6.0 m H perimeter service drifts will be driven to the north and south 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 conveyor drifts are located central to the orebody and run along the north-east–south-west axis. The north-eastern conveyor system extends to the lower north-eastern extremity of the orebody and converges onto the western main conveyor decline. The main west decline rock handling system then transports ore to surface via the western main conveyor decline. Conveyor drifts are 7.5 m w x 6.0 m H and are driven at a maximum gradient of ±9.0°. Section 16.5.8.4 covers the rock handling systems in greater detail.

Mining Areas

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.

16.5.6 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 16-years.

Table 16.42 summarises the LOM development and production results.

Table 16.42 Kamoa 1 Mine Life-of-Mine Development and Production Summary

Waste Development	
Lateral (m)	44,752
Lateral (kt)	4,556
Vertical (m)	6,042
Vertical (kt)	337
Production by Mining Method	
Ore Development (m)	45,388
Ore Development (kt)	4,494
Drift-and-Fill (kt)	116,349
Total Ore Production	
Total Ore Development (kt)	4,494
Total Production (kt)	116,349
Total Tonnes (kt)	120,843
Diluted Grade	
TCu (%)	3.74
S (%)	2.48
As (%)	0.0005
Fe (%)	5.83
AsCu (%)	0.35

1. Notes: Vertical development includes boreholes.
2. Stope shapes designed on an NSR cut-off value of US$100/t NSR.

The following conditions were used in developing the LOM schedule:

- Proximity to the main accesses and initial 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.

16.5.6.1 Productivity Rates

Effective Operating Hours

The effective operating hours per shift are summarised in Table 16.43 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 16.43 Kamoa 1 Mine Shift Rotations and Effective Operating Hours Calculations

Shift Cycle	Calculations
Days per Year	360 days
Number of Crews in Rotation	3
Shifts per Day	2
Shift Duration	12 h
Travelling Time – In	19.50 min / 0.32 h
Travelling 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, and external consultants) and compared with inputs. The cycles were updated accordingly following team discussions. Mine productivities and schedule are based on a development rate of 120 m/month for all primary development.

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

The production rate in ore tonnes per month is highly variable as it is an outcome of the estimated development advance rate, combined with ore thickness, dilution parameters, and ore density.

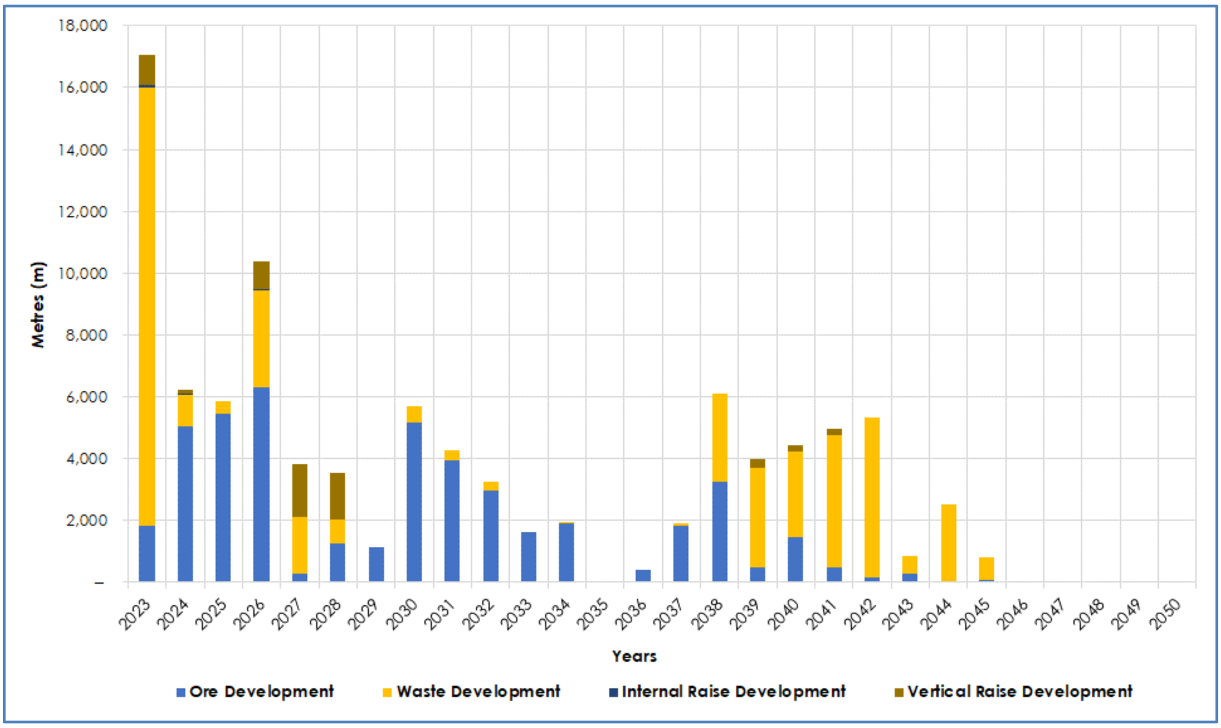
To determine the drift-and-fill production rates, development rates for primary, secondary, and tertiary drifts were combined with paste filling, cable bolting, and end-of-shift blasting restrictions in a block configuration to determine the net block production rate for use in the schedule. The net block production rate was then applied to the drift-and-fill mining shapes within the schedule. The production rate was adjusted depending on the height of the production heading in order to better represent the change in condition. Paste fill barricade construction/placement as well as installation of split sets (geotechnical supports) was not considered in the cycle calculations as it will be completed off critical task.

16.5.6.2 Development Schedules

Development schedule

The life-of-mine development schedule targets the areas required to support the LOM plan. This includes excavating the declines, perimeter service drifts, conveyor drifts, and key infrastructure associated with truck tips, ventilation, dewatering, and maintenance facilities in advance of production areas. Figure 16.65 illustrates the development metres associated with the LOM activities.

Figure 16.65 Kamoa 1 Mine Life-of-Mine Development Schedule



OreWin, 2023.

16.5.7 Mine Production Plan and Scheduling

As the Kamoa Phase 3 mill will be fed by material from Kamoa 1, Kansoko Sud, and Kamoa 2, a balance is formed between the three deposits. Kamoa 1 will initially ramp-up and maintain a production rate of ~2.5 Mtpa for five-years (2023–2029). The remainder of the Kamoa Phase 3 Processing Plant requirements (initially 5 Mtpa) will come from Kansoko Sud and Kamoa 2. Table 16.44 presents the annual ramp-up scheduled tonnes for Kamoa 1.

Table 16.44 Kamoa 1 Mine Ramp-Up Scheduled Tonnage

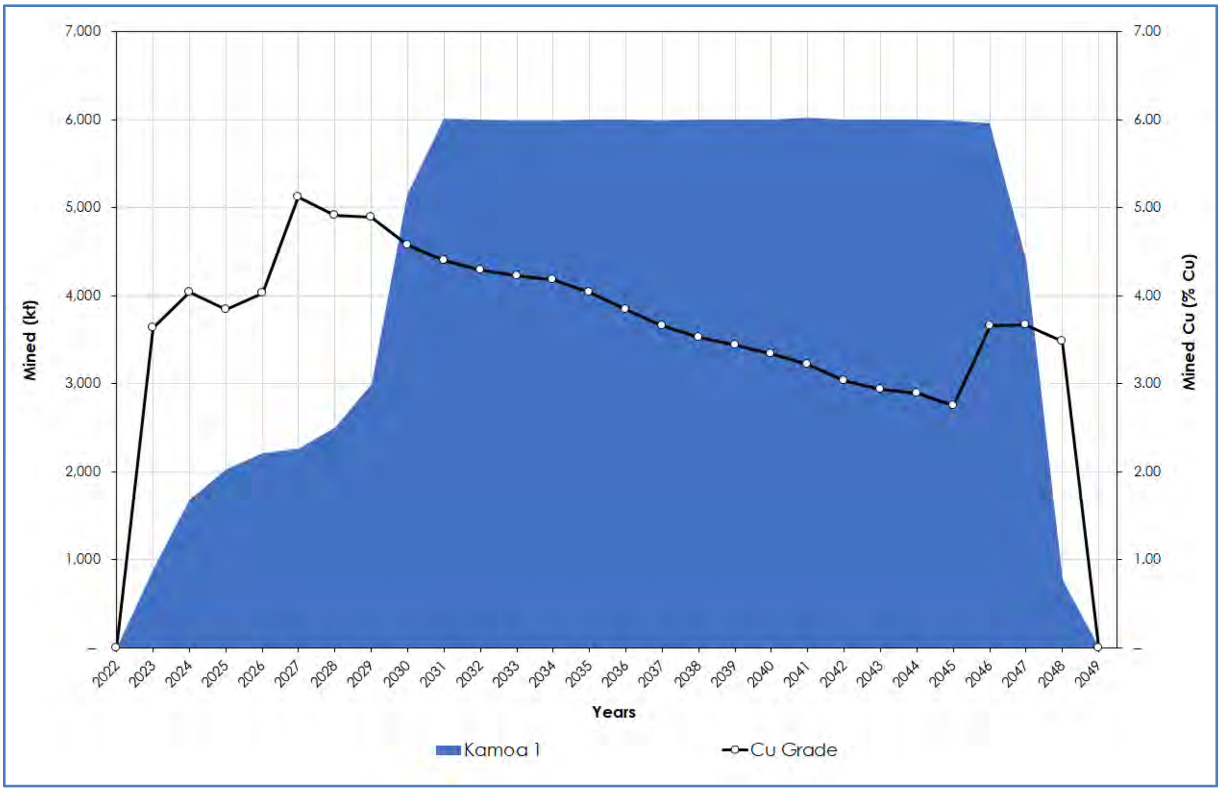
Production Schedule	Years	Scheduled Tonnes (kt)
Ramp-Up (2023)	1	897
Ramp-Up (2024)	1	1,674
Ramp-Up Total	2	2,572

Life-of-Mine Production Schedule

Initially, Kamoa 1 will ramp-up and remain at a steady state of ~2.5 Mtpa for five-years (2023–2029). As the Kamoa Phase 3 Processing Plant increases its rate from 5 Mtpa to 10 Mtpa, Kamoa 1 will ramp-up to its full production rate of 6 Mtpa to meet the milling requirement, with remaining material coming from Kansoko Sud (2 Mtpa) and Kamoa 2 (2 Mtpa). Kamoa 1 will maintain a steady state production rate of 6 Mtpa (2030–2046) until the deposit is depleted and the production rate begins to taper off (2047–2048).

The mining blocks are scheduled so that a higher TCu value is achieved early in the mine life. Figure 16.66 illustrates the LOM production schedule and copper grade.

Figure 16.66 Kamoa 1 Mine Life-of-Mine Production Schedule and Copper Grade



OreWin, 2023.

16.5.8 Underground Infrastructure

16.5.8.1 Mine Ventilation System - Kamoa 1

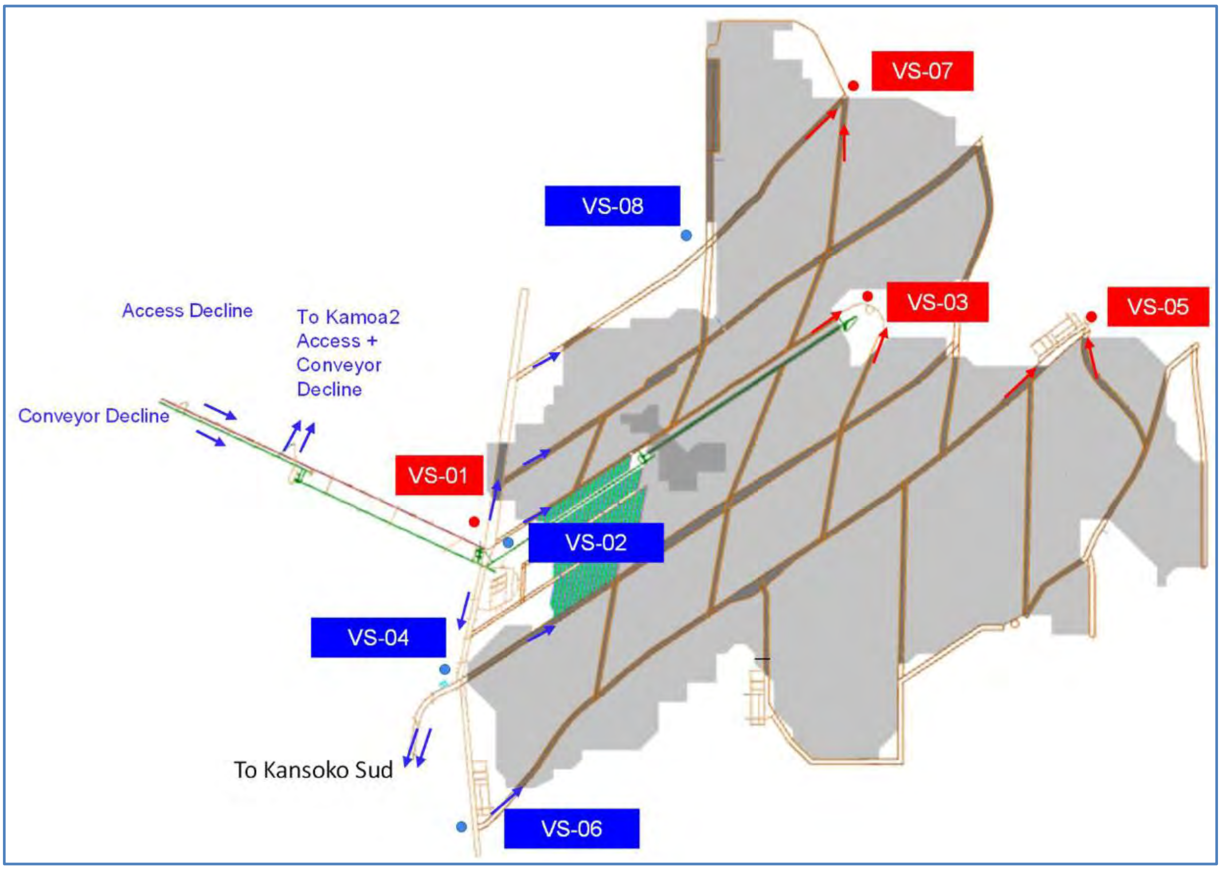
The underground mobile equipment fleet is diesel powered, and mine air cooling is required to maintain underground working air quality within the limits appropriate.

The following assumptions were considered in the ventilation design to maintain safe operating conditions underground and to abide by applicable legislative requirements. Australian and 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.
- Primary and secondary leakage rates used for preliminary airflow estimates for when the mine is fully developed are 10% and 20% respectively. These factors are used 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 30 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.
- Heat load factors for diesel equipment have been split into four types. The following types of trucks, loaders, auxiliary and supporting equipment have respective heat load factors of 1.1, 1.0, 0.8, and 0.3, applied as linear activity tracks in the Ventsim modelling.
- Diesel engines are assumed to have a power conversion efficiency of 35%.

The mine layout schematic illustrating the location of the ventilation shafts is shown in Figure 16.67. The mine will be supplied with fresh air from 4 intake ventilation shafts located to the west of the orebody. The ventilation air will naturally flow through the perimeter drives connecting the west and east of the orebody. The ventilation will be extracted via three exhaust shafts east of the mine. VS-01 will ventilate the conveyor belt directly to return. 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 16.67 Kamoa 1 Mine Layout with Ventilation Shaft Locations



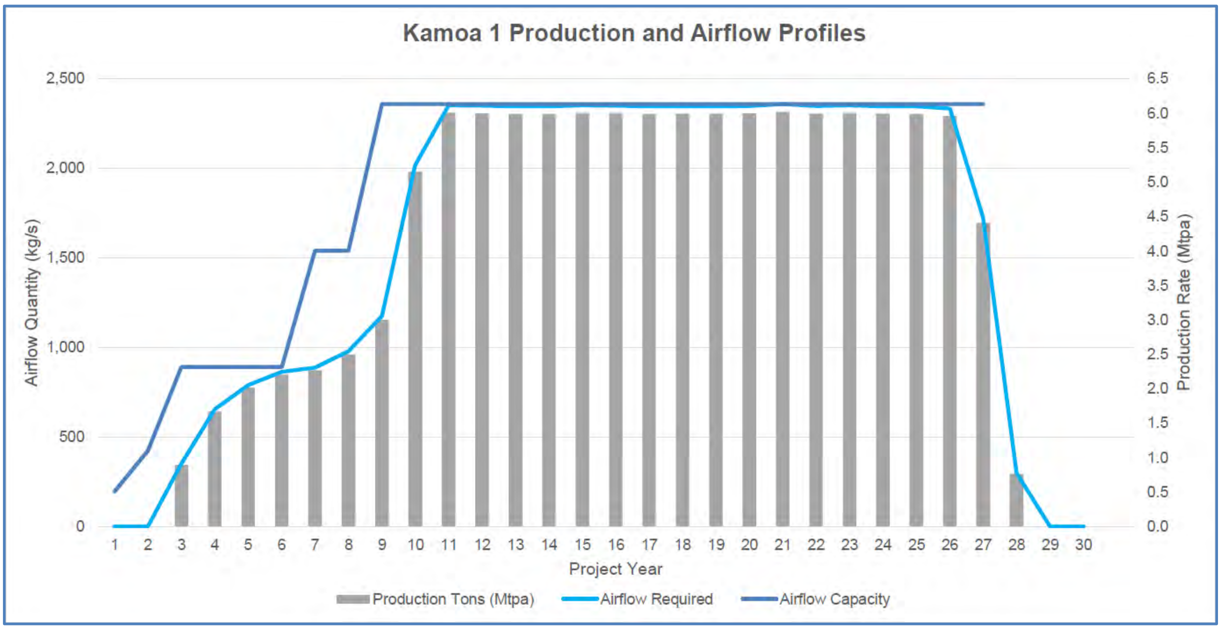
OreWin, 2023.

A summary of the primary ventilation fans is provided in Table 16.45. The LOM airflow requirements are shown in Figure 16.68.

Table 16.45 Kamoa 1 Mine Primary Main Ventilation Fan Requirements

Raise Location	No. of Fans (in parallel)	Operating Range (kg/s)	Peak Airflow (kg/s)	Peak Total Pressure at Collar) (kPa)	Estimated Power (kW)
VS-01	2	220 - 440	220	0.7 – 1.1	380 - 1,100
VS-03	3	660 - 750	678	1.1 – 1.5	910 – 1,610
VS-05	3	660 - 750	678	1.1 – 1.5	910 – 1,610
VS-07	3	660 - 750	678	1.1 – 1.5	910 – 1,610

Figure 16.68 Kamoa 1 Mine Life-of-Mine Airflow Requirements



OreWin, 2023.

The model shows the primary ventilation requirements of 2,340 kg/s at the peak production rate of 6.0 Mtpa and a maximum depth of 1,170 m.

16.5.8.2 Mine Air Cooling Facilities – Kamoa 1

The cooling requirements were estimated from first principles and confirmed by VentSim models which considered auto-compression, surrounding and broken rock heat, vehicle heat, and all other heat components. The model indicated that no mechanical air cooling will be required for Kamoa 1, as fresh air from surface can ventilate the mine adequately at the required depth of 1,170 m below surface. The ventilation system will need to operate at full capacity to provide additional air cooling capacity from surface. Service water will provide cooling capacity and need to be verified during operations as the cooling duty can be overstated at this depth. The last four-years of the mine life will start to exceed thermal design wet-bulb temperature of 27.5°C.

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 16.46.

Table 16.46 Kamoa 1 Mine Auxiliary Ventilation Fan Requirements

Location	Flow per Fan (m^3/s)	Fan Total Pressure (Pa)	Fan Size diameter (m)	Duct Type	Estimated Power (kW)
Development Headings	36	2,300	1.40	Flexible	110
Drift-and-fill Headings	36	2,300	1.40	Rigid and Flexible	110

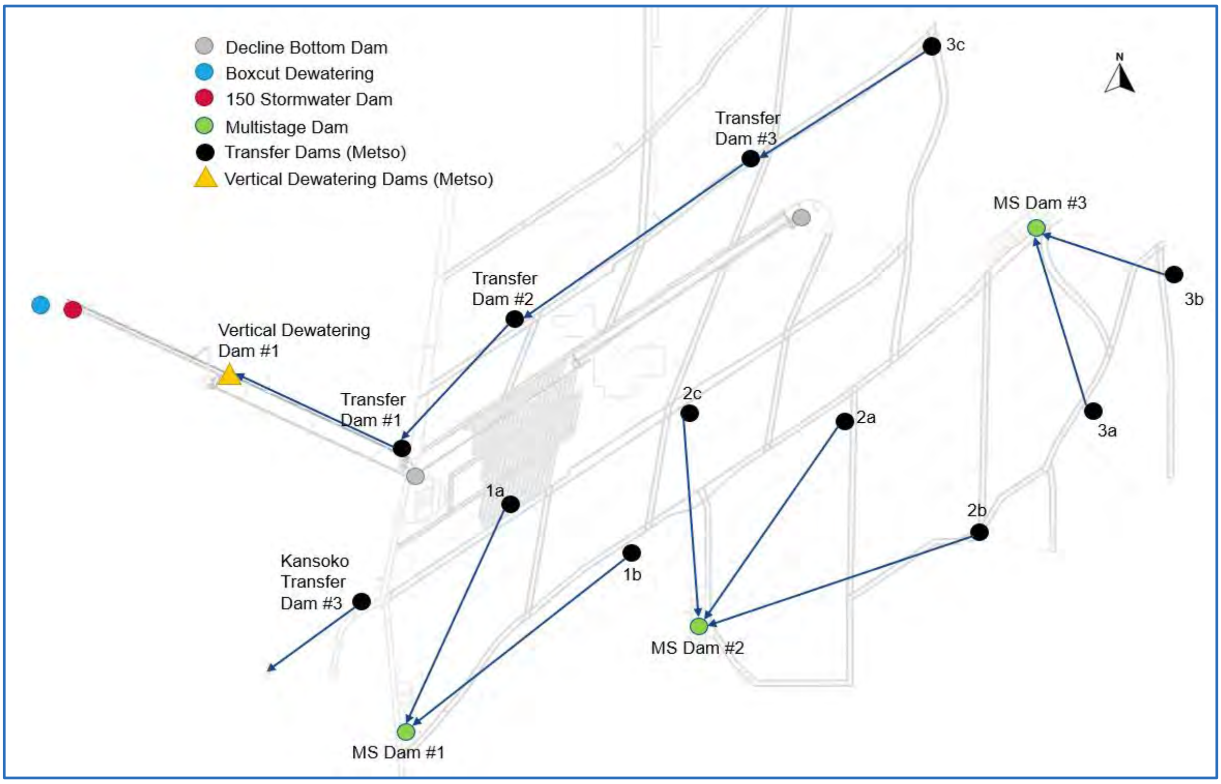
16.5.8.3 Mine Dewatering

Kamoa 1 will have a main decline pumping system which consists of a shaft bottom transfer dam pumping up to an intermediate dam. The intermediate dam is a vertical dewatering dam which will pump out to surface via dewatering boreholes. The main decline pumping arrangement is capable of handling 200 – 1,000 l/s.

In addition to the main decline pumping system, three additional multistage dewatering pump stations will be located as close to the central passage as possible along the dip of the Kamoa 1 orebody – indicated by the green dots in Figure 16.69. Each multistage dewatering pump station will be capable of pumping 1,500 l/s to surface through dewatering boreholes. Kamoa #1will have an expected peak water inflow rate of 3,000 l/s.

Each multistage dewatering pump station will be fed by two or three transfer dam pump stations. The transfer dam pump stations, indicated as black dots in Figure 16.69, will have the capacity to pump 200 – 1,000 l/s to the main multistage dewatering dams. The transfer dams will be strategically positioned to cover the required footprint of the mined ore body.

Figure 16.69 Kamoa 1 Mine Dewatering Layout



DRA, 2022.

16.5.8.4 Rock Handling

The underground rock handling system at Kamoa 1 consists of a Main Decline conveyor which is fed from a main East conveyor. Figure 16.70 highlights the underground conveyor routing in Kamoa 1.

Figure 16.70 Kamoa 1 Mine Underground Rock Handling Conveyor Routing

DRA, 2022.

The main decline rock handling system consists of two belts from shaft bottom to surface. A transfer silo is positioned at the access intersection to the Kamoa 2 orebody. The decline conveyor between shaft bottom and the transfer silo is designed at 2,000 tph, whilst the decline conveyor between the transfer silo and surface is designed for 3,000 tph to cater for additional trucked rock coming out of Kamoa 2.

At the Kamoa 1 / Kamoa 2 intersection, a truck tip and conveyor transfer silo will be constructed. Whilst the truck tip will be constructed during Phase 3, the conveyor transfer will be installed later in the life-of-mine when the Kamoa 2 North South conveyor is installed.

The mining surface rock handling system will receive ore from underground via the main decline conveyor. The system has been designed for a 3,000 t/h capacity to align with the capacity of the system feeding it.

A bypass stockpile feed conveyor will be located on surface equipped with a reversable conveyor at its end that will create two stockpiles. These stockpiles will be loaded by FEL and hauled by trucks to the stockpile area. Reclaiming will be done by FEL and haul trucks tipping into a bulk reclaim tip equipped with rock breaker, apron feeder and belt magnet.

16.5.8.5 Materials Handling Logistics and Storage

The material handling procedures for Kamoa 1, will be the same as those for Kakula. Men and materials will be transported underground via the service decline, while emulsion, shotcrete, fuel, and lube will be transferred underground using boreholes. These facilities will be established in the vicinity of the main underground workshops.

16.5.8.6 Workshops

Kamoa 1 will have one centrally located main workshop and four satellite workshops spread out to cover the footprint of the mining area. Figure 16.71 below indicates the approximate positions for the main and satellite workshops.

Figure 16.71 Kamoa 1 Mine Workshops

DRA, 2022.

16.5.8.7 Fuel and Lubricant Distribution

Six new gensets are planned for Kamoa 1 and will be supplied with 6 x 70 m³ diesel tanks and two 4.5 m³ oil storage tanks. Kamoa 1 will have two surface and two underground workshops. The two surface workshops each will require one 70 m³ diesel tank and four oil storage tanks. One main underground workshop is planned and will require the normal storage of three 15 m³ diesel storage tanks and four oil storage tanks. One 4.5 m³ batch tank and three 2.5 m³ batch tanks for each of the underground workshops will be supplied for the storage and supply of hydrocarbons to the underground workings.

The total number of tanks planned for Kamoa 1 will be forty-one.

16.5.8.8 Toilet System

As described under Kakula Mine.

16.5.8.9 Electrical Power Requirements

The electrical power requirements for the additional equipment for all areas at Kamoa 1 are indicated in Figure 16.72 and Table 16.47.

Figure 16.72 Kamoa 1 Mine Life-of-Mine Maximum Demand Profile

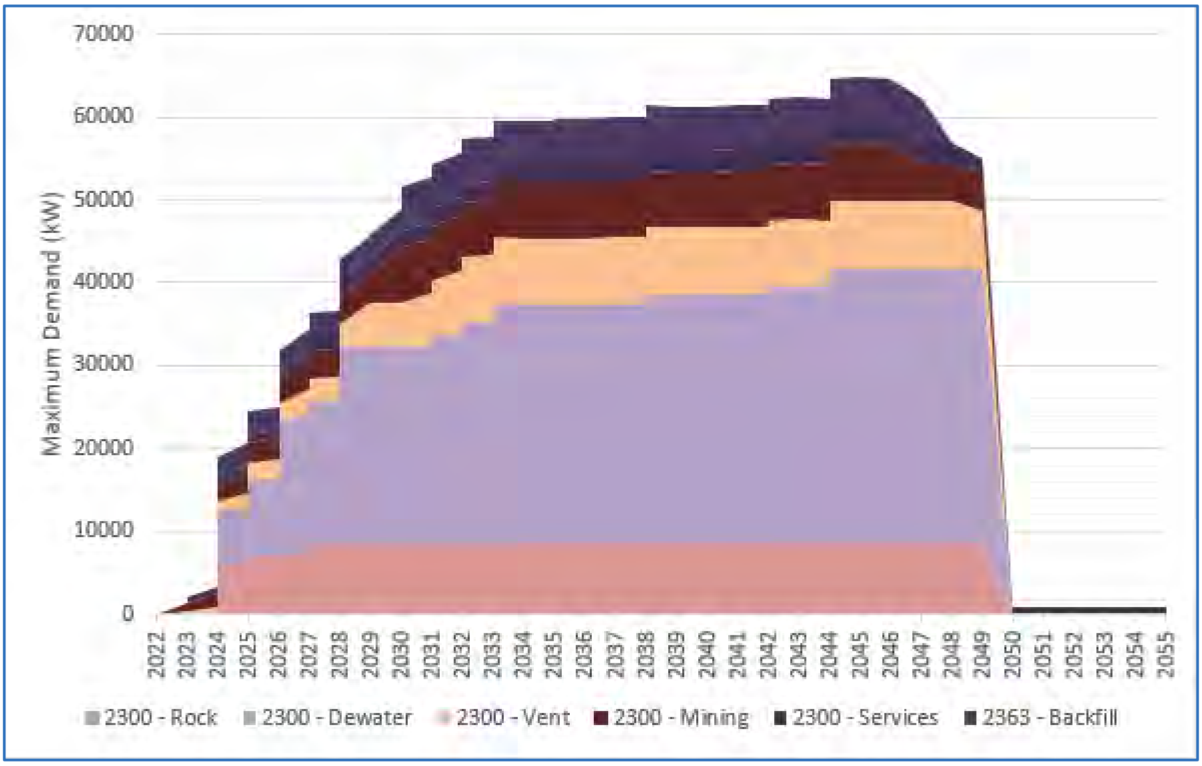


Table 16.47 Kamoa 1 Mine Life-of-Mine Maximum Demand

WBS - Area	Maximum Demand (kW) (2046)
2300 - Rock	8,620
2300 - Dewater	33,161
2300 - Vent	8,126
2300 - Mining	6,800
2300 - Services	2,562
2300 - Services	2,562
2363 - Backfill	5,400
Total	67,233

16.5.8.10 Power Distribution

A new 220 kV yard and 33 kV infrastructure will be installed at Kamoa. This 33 kV infrastructure will distribute power via 33 kV overhead lines and cables to the various areas as required within Kamoa Complex.

The Kamoa 1 Portal 33 kV substation is fed via overhead line from the 33 kV Central substation. Three 30 MVA 33/11 kV transformer supply power to the Portal 11 kV substation.

A 33/11 kV substation is installed at the Kamoa 1 portal.

Surface ventilation shaft equipment, refrigeration plants and backfill plant are fed via a dual redundant 33 kV overhead line to a transformer.

Underground power distribution will be via 11 kV and underground 11 kV distribution substations. For smaller loads, the mini substations will provide suitable LV supplies (690 V, 400 V, 1,000 V), as required in the area.

The 33 kV distribution substations and 11 kV distribution substations allowed for are listed below. Power is distributed from these substations to various equipment substations.

- K1M 33 kV Kamoa 1 Portal Substation
- K1M 11 kV Kamoa 1 Portal Substation

16.5.9 Equipment

All equipment is sized for a 6.0 Mtpa case to support a drift-and-fill mining method. 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 based on the existing fleet and cover the major components for the operation.

16.5.9.1 Mobile Equipment

The mobile equipment is diesel-powered, rubber-tyred. 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.

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 for Kamoa 1 is listed in Table 16.48.

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Table 16.48 Kamoa 1 Mine Mobile Equipment List

Description	Maximum Number Required	Number of Units to Purchase	Number of Rebuilds
Drill Rig 282	22	38	30
LHD – 21 t	22	43	34
Haul Truck – 63 t	23	35	27
Concrete / Shotcrete Mixer Truck	10	10	N/A
Shotcrete Sprayer	10	15	N/A
Scissor Lift	15	42	N/A
Charmec – Explosives Loading Truck	10	37	N/A
Mine Support Equipment			
Explosives Transport Truck	10	39	N/A
Agicar – Concrete Mixing Truck – 12 m^3	4	4	N/A
Shotcrete – Backfill	4	10	N/A
Scissor Lift – Backfill	10	28	N/A
LVs	55	160	N/A
Personnel Carriers	13	38	N/A
Grader	3	12	N/A
Utility Equipment – Material	9	36	N/A
Utility Equipment – Maintenance	7	28	N/A
Telehandlers	10	39	N/A
Underground Mobile Crane	3	12	N/A
Skidsteer	7	9	N/A

16.5.9.2 Fixed Equipment

As described under Kakula, Section 16.3.9.2.

16.6 Kansoko Sud Underground Mining

16.6.1 Introduction

Based on updated design criteria, the mining method, mine design, and production schedule have been updated from previous studies. Mining method 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 selected and the resultant mine designs and schedules.

The mining method used for the Kansoko Sud deposit is drift-and-fill. The Kansoko Sud Mineral Reserve by mining method is summarised in Table 16.49.

Table 16.49 Kansoko Sud Mine Probable Mineral Reserves by Mining Method

Production by Mining Method	Ore (Mt)	TCu (%)	Fe (%)	As (%)	S (%)	AsCu (%)
Ore Development	1.15	3.62	6.40	0.00	2.55	0.22
Drift-and-Fill	36.65	3.71	6.18	0.00	2.66	0.23
Total Ore*	37.81	3.70	6.19	0.00	2.65	0.23

*May not sum to total due to rounding.

16.6.2 Mine Design Parameters

16.6.2.1 Ore and Waste Properties

The Kansoko Sud deposit is a large stratiform copper deposit, typical of sediment hosted deposits. The deposit is tabular, with dips varying from 0–35° with an average dip of 17°. The thicknesses varies from 3 m to 6 m. The ore zone density has been defined as using a greater than 2.4% Cu (total copper grade) cut-off. The swell factor for development is 50%.

Table 16.50 details the bulk density parameters of the ore and surrounding waste rock of the Kansoko Sud deposit.

Table 16.50 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

16.6.3 Mine Planning

16.6.3.1 Lateral Development

The northern declines have a maximum gradient of ±9.0°. The conveyor drift is 7.5 m W x 6.0 m H with 1.5 m arch corners, and 6 m W x 6.0 m H with 1.5 m arch corners for the service drift. 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 6 m w x 6 m H, with 1.5 m arch corners unless otherwise specified.

Conveyor drifts have a maximum gradient of ±9°. They are 7.5 m w x 6 m H (1.5 m arch corners), with re-mucks located every 300 m.

All perimeter drifts are 6 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 crosscuts 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° and 1.5 m arch corners.

16.6.3.2 Vertical Development

Vertical development consists of ventilation raises, bins, and boreholes. All ventilation raises are excavated with a raisebore drill. All ventilation shafts are designed at 6 m in diameter. Access drifts to the ventilation shafts are 6 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.

16.6.3.3 Drift-and-fill

The primary mining method for the Kamoa-Kakula deposits (drift-and-fill) was selected based on the results of a trade-off study which compared six different variations of drift-and-fill mining to identify the most suitable. The selected drift-and-fill mining method is explained in detail in Section 16.2.2.

A typical mining block with production drifts perpendicular to the connection drift is 198 m wide. Each mining block consist of three mining units. Each mining unit comprises three primary headings, three secondary headings, and three tertiary headings.

The production drift cross-sectional shape differs with the type of production heading. Typical primary and secondary production drifts have an 8 m wide arch (1.5 m arch corners), with a maximum height up to 7.6 m. Slipe drifts are a 6 m wide arch (1.5 m arch corners), with a maximum height up to 7.6 m.

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 (9°).

16.6.4 Backfill

Paste fill will be the primary backfilling strategy for the Kansoko Sud 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 central services drifts. Distribution pipes installed in the connection drifts will then deliver the paste fill to the production areas.

16.6.5 Mine Access Design

Box-Cuts

There is one box-cut developed for access to the underground workings. The northern box-cut incorporates two portals.

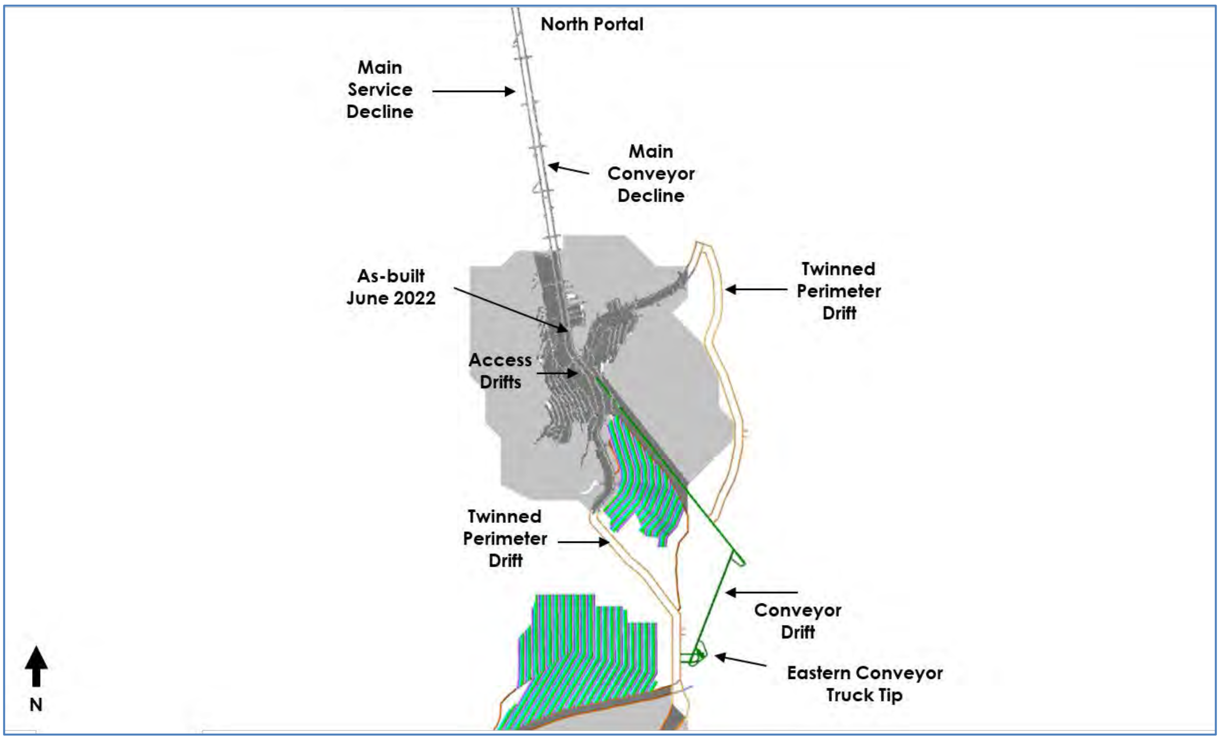
Main Declines

The deposit is accessed via a twinned decline on the northern side.

One of the northern declines is the primary mine service access, and the other decline is a conveyor haulage drift. The service decline has dimensions of 6.0 m w x 6.0 m H, with the conveyor decline 7.5 m w x 6.0 m H. Both northern declines have a maximum gradient of ±9.0° gradient. Development of the declines has been completed as shown in Figure 16.73.

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. Travel direction in the declines is uphill in the main conveyor decline and downhill in the main service decline. Approximately 1,150 m from the portal, access drifts are developed off the main service decline to the perimeter declines. These drifts also provide access to the initial truck tip area, as shown in Figure 16.73.

Figure 16.73 Kansoko Sud Mine Main Access Development



OreWin, 2023.

Perimeter Service and Conveyor Drifts

From the bottom of the north decline, 6.0 m w x 6.0 m H service drifts are developed through the middle of the orebody to the southern extents of the deposit. There are two sets of perimeter service drifts that enable access to the outer sections of the deposit, one in the north-east, and the other in the south-west. This development will serve as the primary accesses to the production areas and underground infrastructure. These service 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 conveyor drifts are located central to the orebody and extend to the upper eastern side of the orebody. The eastern conveyor system extends to the top of the southern section of the orebody and converges onto the northern main conveyor decline as shown in Figure 16.73. The main north decline rock handling system then transports ore to surface via the northern main conveyor decline. Conveyor drifts are 7.5 m w x 6.0 m H and are driven at a maximum gradient of 9.0°. Section 16.6.8.4 covers the rock handling systems in greater detail.

Mining Areas

For drift-and-fill mining, connection drifts will be developed towards the eastern and western extents of the orebody from the central service drifts. These will serve as the main accesses to the production blocks. Connection drifts will provide access and ventilation to the planned mining areas.

16.6.6 Mine Development and Production Schedules

As Kansoko Sud is a current operating mine, the development schedule focuses on continuing to establish mine services and supporting infrastructure to enable the production mining areas to ramp-up to 2.0 Mtpa ore production. Based on a 360-day operating schedule, the production goal is to sustain full production for 14-years.

Table 16.51 summarises the LOM development and production results.

Table 16.51 Kansoko Sud Mine Life-of-Mine Development and Production Summary

Waste Development	
Lateral (m)	12,442
Lateral (kt)	1,266
Vertical (m)	1,632
Vertical (kt)	100
Production by Mining Method	
Ore Development (m)	11,917
Ore Development (kt)	1,154
Drift-and-Fill (kt)	36,658
Total Ore Production	
Total Ore Development (kt)	1,154
Total Production (kt)	36,658
Total Tonnes (kt)	37,813
Diluted Grade	
TCu (%)	3.70
S (%)	2.65
As (%)	0.0014
Fe (%)	6.19
AsCu (%)	0.23

1. Notes: Vertical development includes boreholes.
2. Stope shapes designed on an NSR cut-off value of US$100/t NSR.

The following conditions were used in developing the LOM schedule:

- Proximity to the main accesses and initial 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.

16.6.6.1 Productivity Rates

Effective Operating Hours

The effective operating hours per shift are summarised in Table 16.52 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 16.52 Kansoko Sud Mine Shift Rotations and Effective Operating Hours Calculations

Shift Cycle	Calculations
Days per Year	360 days
Number of Crews in Rotation	3
Shifts per Day	2
Shift Duration	12 h
Travelling Time – In	19.50 min / 0.32 h
Travelling 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, and external consultants) and compared with inputs. The cycles were updated accordingly following team discussions. Mine productivities and schedule are based on a development rate of 120 m/month for all primary development.

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

The production rate in ore tonnes per month is highly variable as it is an outcome of the estimated development advance rate, combined with ore thickness, dilution parameters, and ore density.

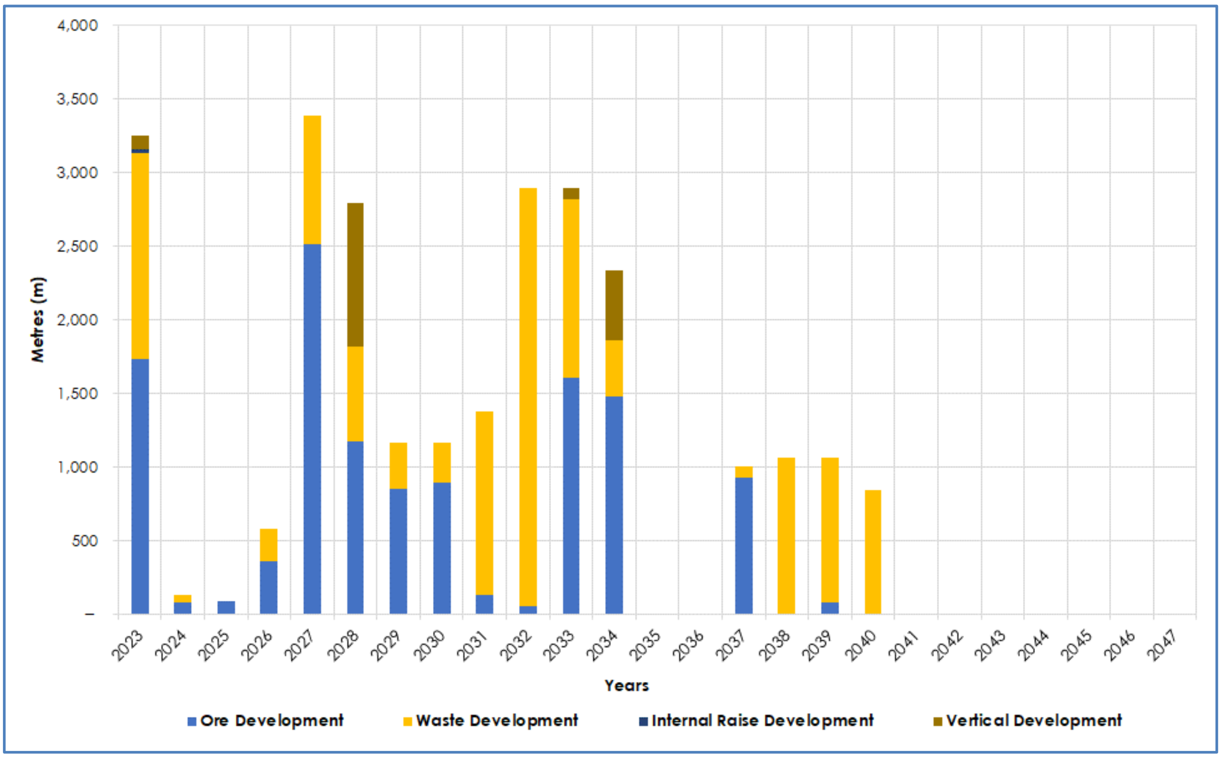
To determine the drift-and-fill production rates, development rates for primary, secondary, and tertiary drifts were combined with paste filling, cable bolting, and end-of-shift blasting restrictions in a block configuration to determine the net block production rate for use in the schedule. The net block production rate was then applied to the drift-and-fill mining shapes within the schedule. The production rate was adjusted depending on the height of the production heading in order to better represent the change in condition. Paste fill barricade construction/placement as well as installation of split sets (geotechnical supports) was not considered in the cycle calculations as it will be completed off critical task.

16.6.6.2 Development Schedules

Development schedule

The life-of-mine development schedule targets the areas required to support the LOM plan. This includes excavating the perimeter service drifts, conveyor drifts, and key infrastructure associated with truck tips, ventilation, dewatering, and maintenance facilities in advance of production areas. Figure 16.74 illustrates the development metres associated with the LOM activities.

Figure 16.74 Kansoko Sud Mine Life-of-Mine Development Schedule



OreWin, 2023.

16.6.7 Mine Production Plan and Scheduling

As the Kamoa Phase 3 Processing Plant will be fed by material from Kamoa 1, Kansoko Sud, and Kamoa 2, a balance is formed between the three deposits. Kansoko Sud will initially ramp-up and maintain a production rate of ~1.3 Mtpa for five-years (2023–2029). The remainder of the Kamoa Phase 4 Processing Plant requirements (initially 5 Mtpa) will come from Kamoa 1 and Kamoa 2. Table 16.53 presents the annual ramp-up scheduled tonnes for Kansoko Sud.

Table 16.53 Kansoko Sud Mine Ramp-up Scheduled Tonnage

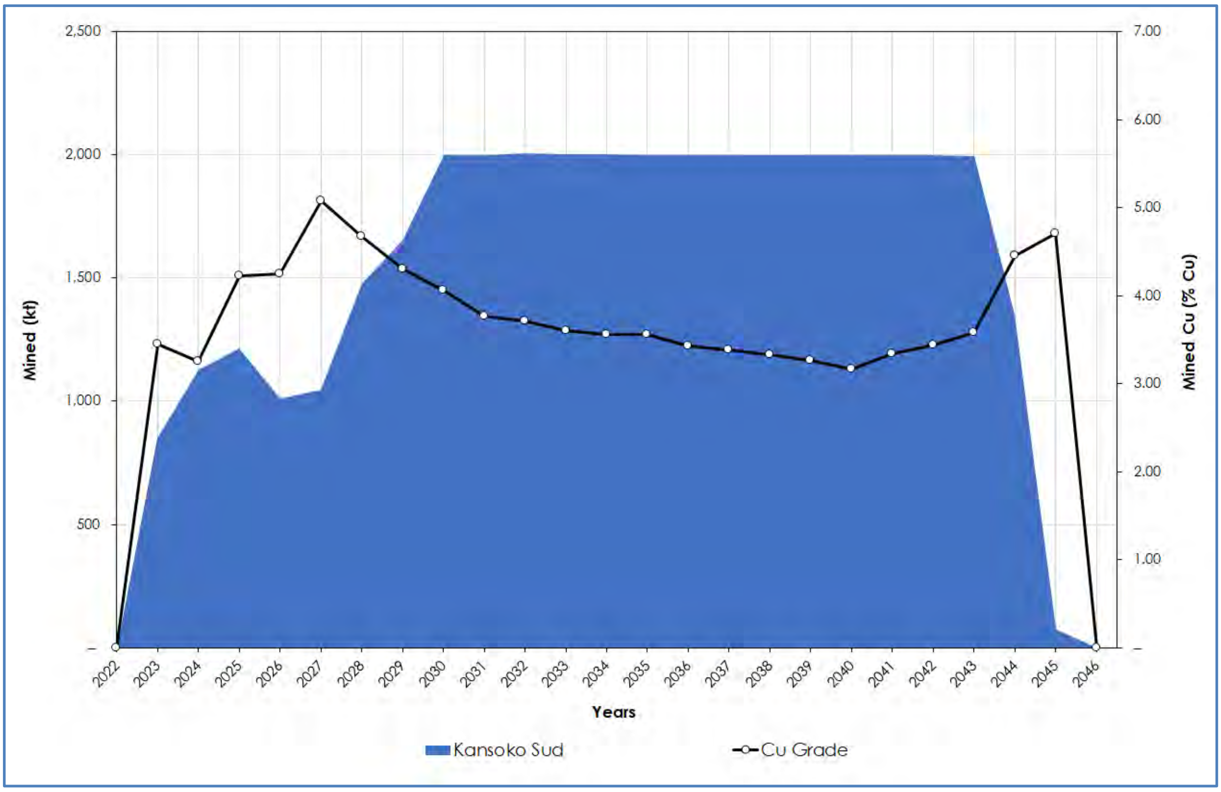
Production Schedule	Years	Scheduled Tonnes (kt)
Ramp-Up (2023)	1	851
Ramp-Up (2024)	1	1,129
Ramp-Up Total	2	2,264

Life-of-Mine Production Schedule

Initially, Kansoko Sud will ramp-up and remain at a steady state of ~1.3 Mtpa for five-years (2023–2029). As the Kamoa Phase 3 Processing Plant increases its rate from 5 Mtpa to 10 Mtpa, Kansoko Sud will ramp-up to its full production rate of 2 Mtpa to meet the milling requirement, with remaining material coming from Kamoa 1 (6 Mtpa) and Kamoa 2 (2 Mtpa). Kansoko Sud will maintain a steady state production rate of 2 Mtpa (2030–2043) until the deposit is depleted and the production rate begins to taper off (2044–2045).

The mining blocks are scheduled so that a higher TCu value is achieved early in the mine life. Figure 16.75 illustrates the LOM production schedule and copper grade.

Figure 16.75 Kansoko Sud Mine Life-of-Mine Production Schedule and Copper Grade



OreWin, 2023.

16.6.8 Underground Infrastructure

16.6.8.1 Mine Ventilation System - Kansoko Sud

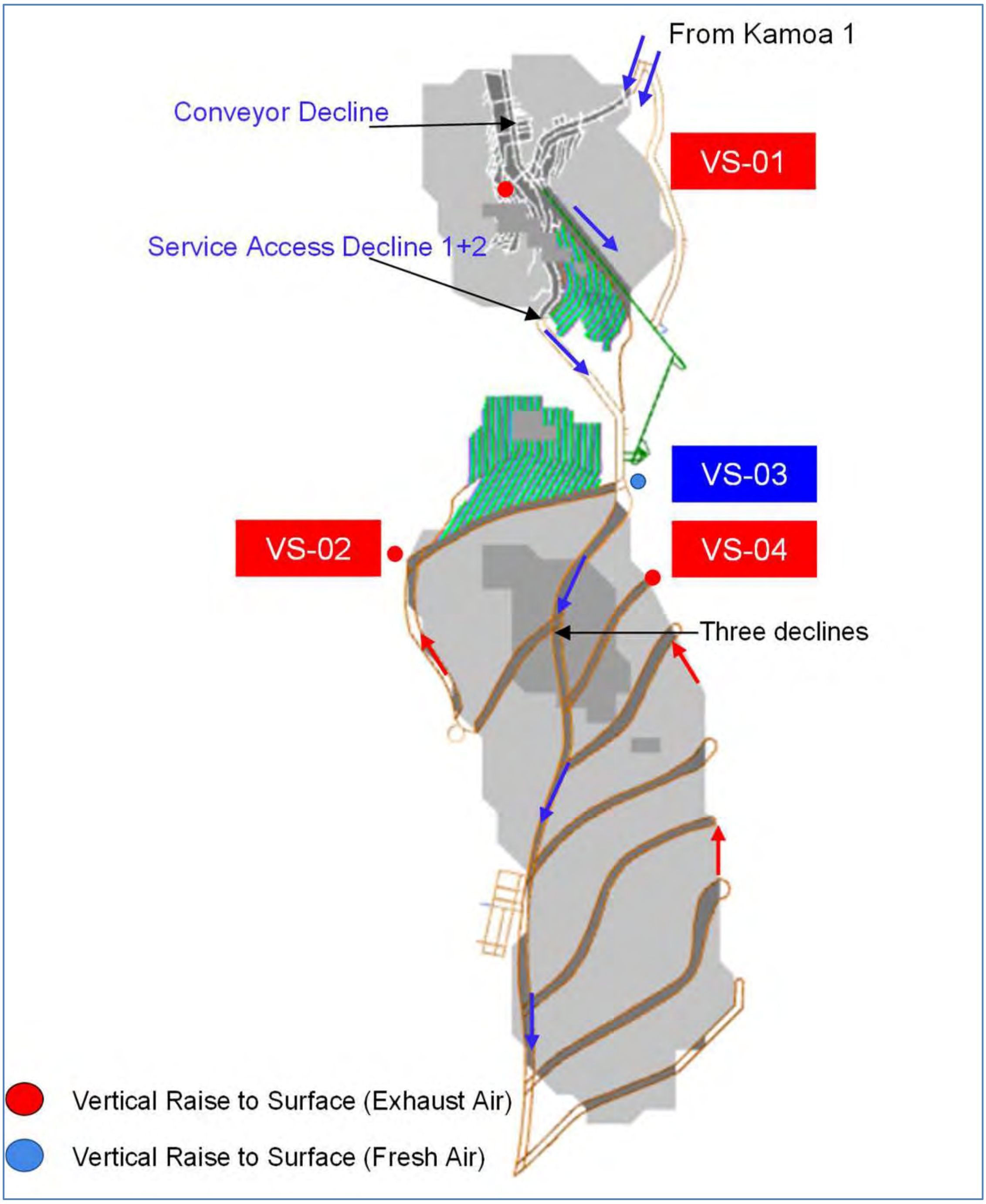
The underground mobile equipment fleet is diesel powered, and mine air cooling is required to maintain underground working air quality within the appropriate limits.

The following assumptions were considered in the ventilation design to maintain safe operating conditions underground and to abide by applicable legislative requirements. Australian and 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.
- Primary and secondary leakage rates used for preliminary airflow estimates for when the mine is fully developed are 10% and 20% respectively. These factors are used 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 30 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.
- Heat load factors for diesel equipment have been split into four types. The following types of trucks, loaders, auxiliary and supporting equipment have respective heat load factors of 1.1, 1.0, 0.8, and 0.3, applied as linear activity tracks in the Ventsim modelling.
- Diesel engines are assumed to have a power conversion efficiency of 35%.

The mine layout schematic illustrating the location of the ventilation shafts is shown in Figure 16.76. The mine will be supplied with fresh air from the decline as well as an intake ventilation shaft located to the east of the orebody. The ventilation air will naturally flow through three perimeter drives connecting the north and south of the orebody and will be extracted via three exhaust shafts. VS-01 will ventilate the conveyor belt directly to return. 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 16.76 Kansoko Sud Mine Layout with Ventilation Shaft Locations



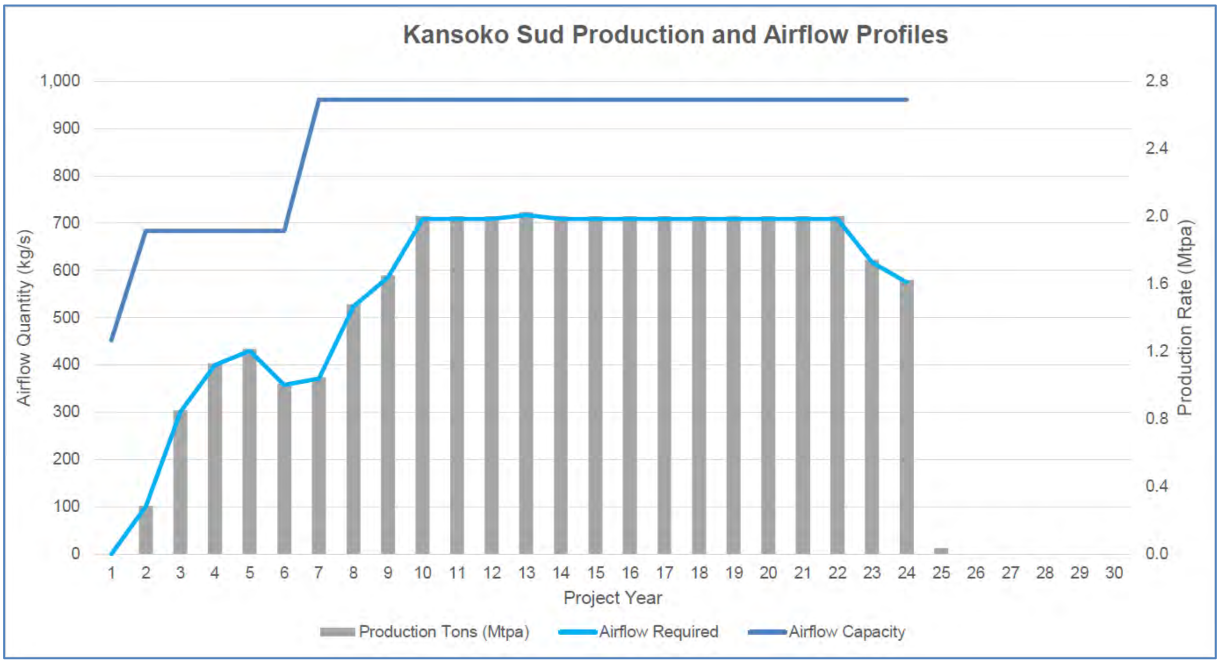
OreWin, 2023.

A summary of the primary ventilation fans is provided in Table 16.54. The LOM airflow requirements are shown in Figure 16.77.

Table 16.54 Kansoko Sud Mine Primary Main Ventilation Fan Requirements

Raise Location	No. of Fans (in parallel)	Operating Range (kg/s)	Peak Airflow (kg/s)	Peak Total Pressure at Collar (kPa)	Estimated Power (kW)
VS-01	2	220 – 440	220	1.2 – 1.5	380 - 1,100
VS-02	2	220 – 500	375	1.2 – 1.8	380 - 1,300
VS-04	2	220 - 500	375	1.2 – 1.8	380 - 1,300

Figure 16.77 Kansoko Sud Mine Life-of-Mine Airflow Requirements



OreWin, 2023.

The model shows the primary ventilation requirements of 750 kg/s at the peak production rate of 2 Mtpa and a maximum depth of 370 m.

16.6.8.2 Mine Air Cooling Facilities – Kansoko Sud

The cooling requirements were estimated from first principles and confirmed by VentSim models which considered auto-compression, surrounding and broken rock heat, vehicle heat, and all other heat components. The model indicated that no mechanical air cooling will be required for Kansoko Sud, as fresh air from surface can ventilate the mine adequately at the required depth of 370 m below surface.

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 16.55.

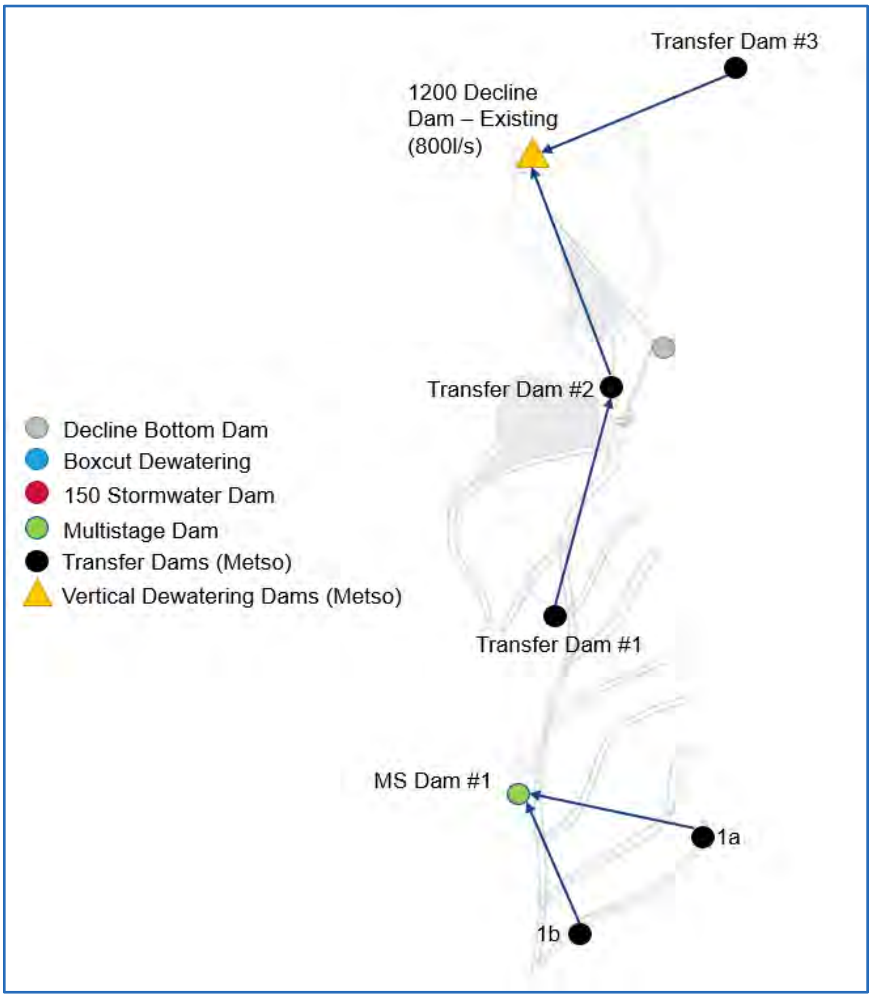
Table 16.55 Kansoko Sud Mine Auxiliary Ventilation Fan Requirements

Location	Flow per Fan (m^3/s)	Fan Total Pressure (Pa)	Fan Size diameter (m)	Duct Type	Estimated Power (kW)
Development Headings	36	2,300	1.40	Flexible	110
Drift-and-fill Headings	36	2,300	1.40	Rigid and Flexible	110

16.6.8.3 Mine Dewatering

Kansoko Sud mine will have a single multistage dewatering dam with two transfer dams feeding it. Kansoko Sud also has existing pump stations which will contribute to the dewatering efforts in the area. During the early phases of development, a transfer pump station will be established at Kamoa 1, and Kansoko which will pump back to the existing Kansoko Sud pump station and out to surface from there. As shown in Figure 16.78. Kansoko will have a peak expected fissure water inflow rate of approximately 650 l/s.

Figure 16.78 Kansoko Sud Mine Dewatering Layout

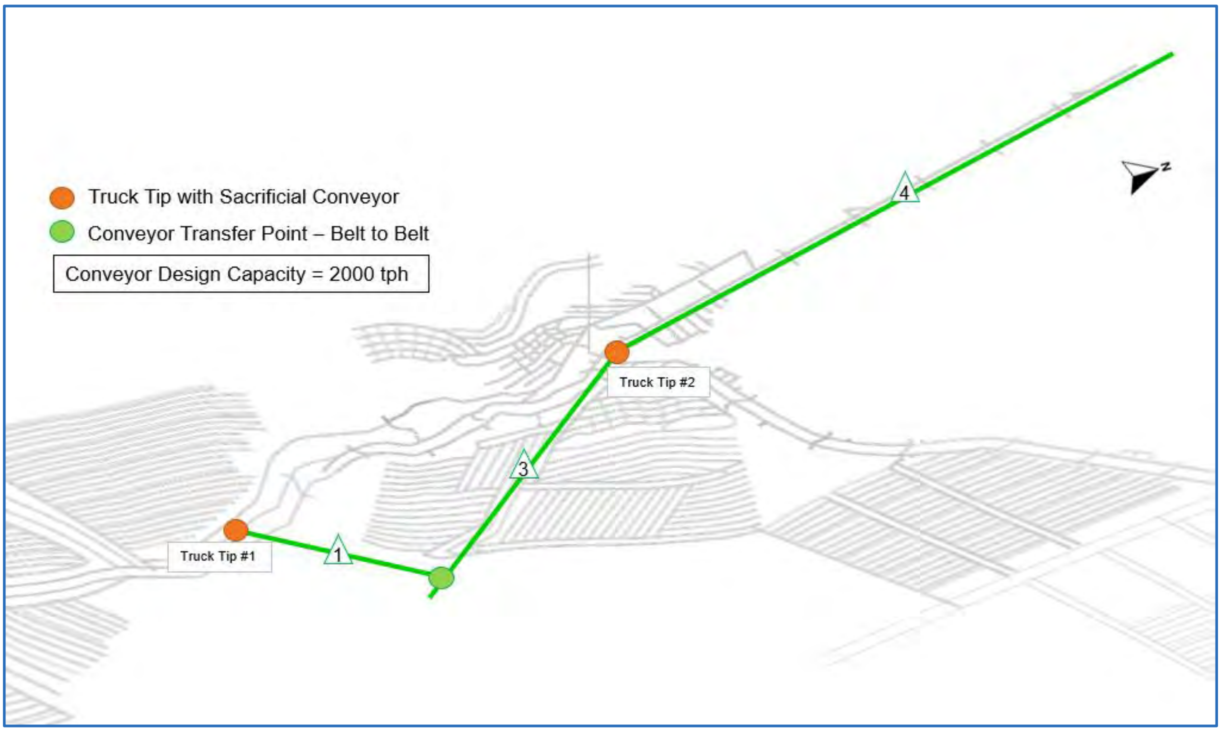


DRA, 2022.

16.6.8.4 Rock Handling

The underground rock handling system at Kansoko Sud is illustrated in Figure 16.79.

Figure 16.79 Kansoko Sud Mine Rock Handling System Overview



DRA, 2022.

Two truck tips will be established underground of similar design as the rest of the operations. Material will be conveyed to surface where material will be transferred onto a short overland conveyor that will link back to the main incoming Kamoa 1 conveyor system feeding the ROM stockpile at the Kamoa Plant. The Kansoko Sud surface rock handling system also includes a bypass discharge, bypass stockpile and bypass bulk reclaim tip that can be used when the Kamoa 1 conveyor is not working or when the system is congested.

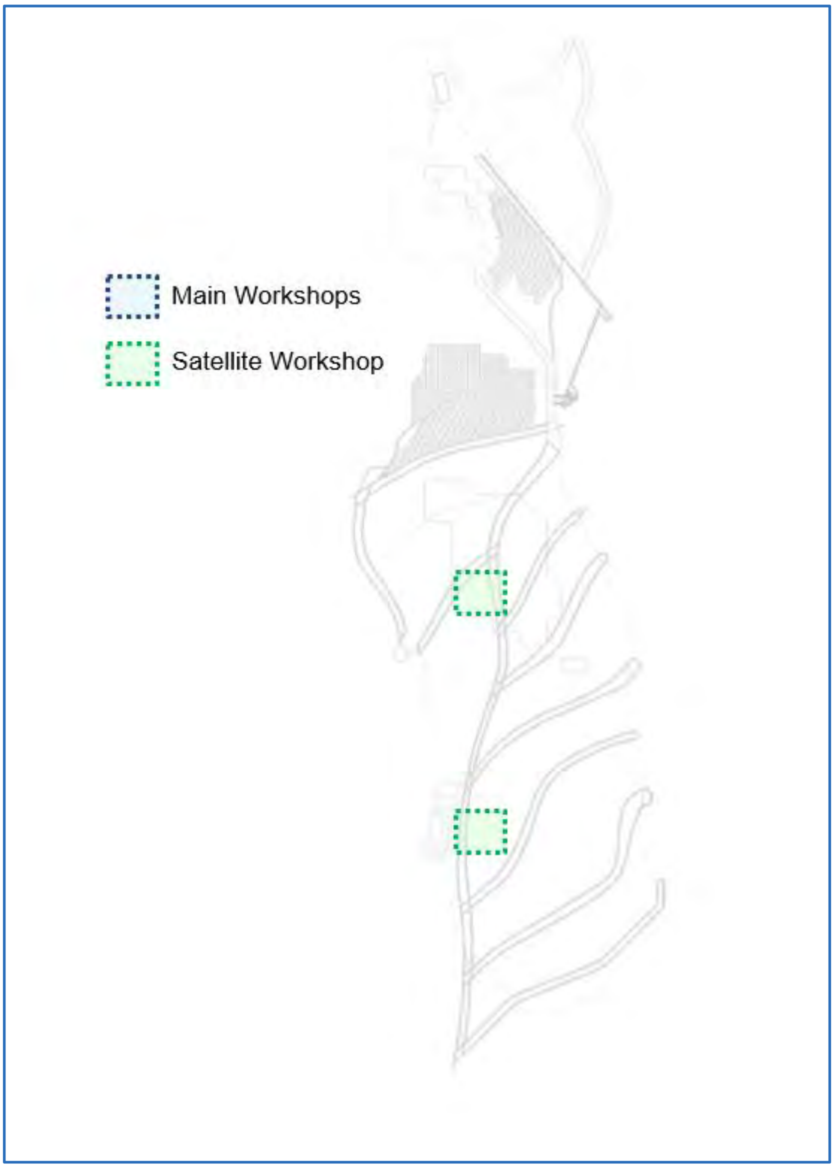
16.6.8.5 Materials Handling Logistics

The material handling procedures for Kansoko Sud will be limited to using UV's, and LDV's, to transport all required material from surface stores, or from the Kamoa 1 main underground workshop to working areas.

16.6.8.6 Workshops

Kansoko Sud will have one centrally located main workshop and two satellite workshops on the southern side of the mining area. Figure 16.80 below indicates the approximate positions for the main and satellite workshops.

Figure 16.80 Kansoko Sud Mine Workshops



DRA, 2022.

16.6.8.7 Fuel and Lubricant Distribution

Kansoko Sud will have two surface and one underground workshops. The two surface workshops each will require one 70 m^3 diesel tank and four oil storage tanks. One underground workshop is planned and will require the normal storage of three 15 m^3 diesel storage tanks and four oil storage tanks. One 4.5 m^3 batch tank, and three 2.5 m^3 batch tanks for the underground workshop will be supplied for the storage and supply of hydrocarbons to the underground workings.

The total number of tanks planned for Kansoko Sud will be twenty-two.

16.6.8.8 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 lubricants, 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.

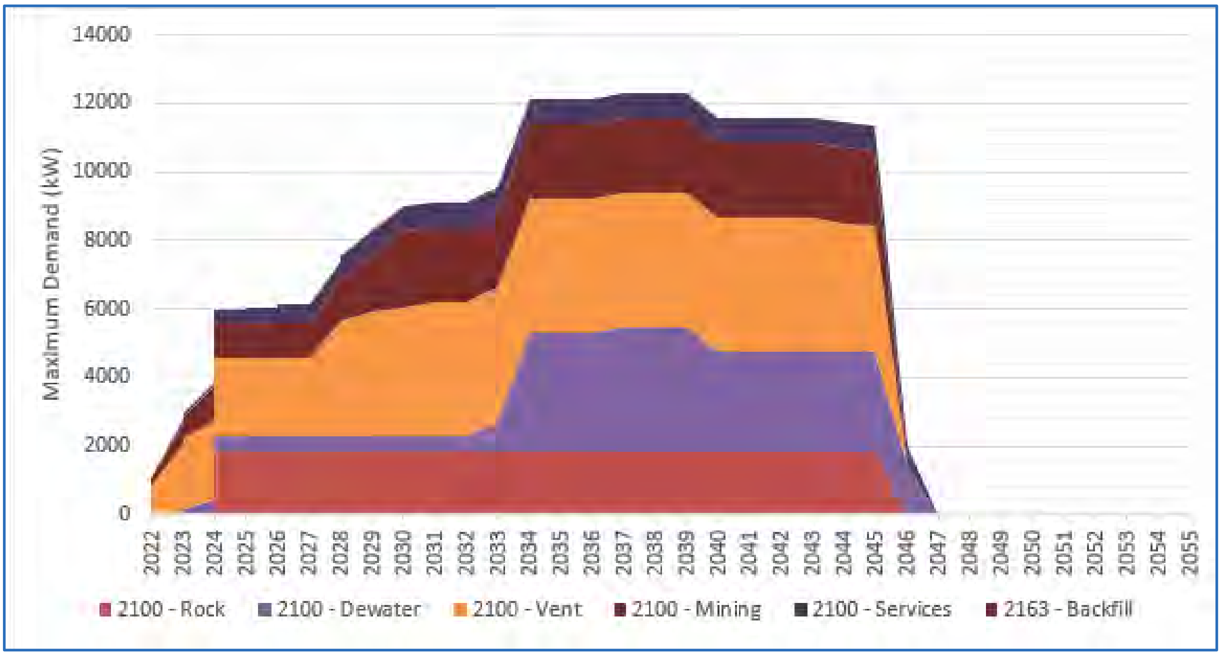
16.6.8.9 Toilet System

As described under Kakula Mine.

16.6.8.10 Electrical Power Requirements

The electrical power requirements for all areas at Kansoko are indicated in Figure 16.81 and Table 16.56 below.

Figure 16.81 Kansoko Sud Mine Life-of-Mine Maximum Demand Profile



OreWin, 2023.

Table 16.56 Kansoko Sud Mine Life-of-Mine Maximum Demand

WBS - Area	Maximum Demand (kW) (2037)
2100 – Rock Handling	1,824
2100 - Dewatering	3,644
2100 – Ventilation and Cooling	3,916
2100 - Mining	2,200
2100 - Services	728
Total	12,313

16.6.8.11 Power Distribution

The new 220 kV yard, and 33 kV infrastructure, that is part of the Kamoa #1 infrastructure, will be used to service Kansoko mine. The 220/33 kV Kamoa#1 KCS will be equipped with three 80 MVA 220/33 kV transformers. This 33 kV infrastructure will distribute power via 33 kV overhead lines and cables to the various areas as required. The Kansoko portal is fed via overhead lines from the 33 kV Central substation.

Underground power distribution will be via 11 kV cable with 11 kV distribution substations. For smaller loads, the mini substations will provide suitable LV supplies (690 V, 400 V, 1,000 V), as required in the area.

The 33 kV distribution substations and 11 kV distribution substations allowed for are listed below. Power is distributed from these substations to various equipment substations.

- KSM 33 kV Kansoko Substation
- KSM 11 kV Kansoko Substation
- KSM 11 kV Kansoko Substation #2
- KSM 11 kV Kansoko Substation #3
- KSM 11 kV Kansoko Substation #4

16.6.9 Equipment

All equipment is sized for a 2.0 Mtpa case to support a drift-and-fill mining method. 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 based on the existing fleet and cover the major components for the operation.

16.6.9.1 Mobile Equipment

The mobile equipment is diesel-powered, rubber-tyred. 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.

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 for Kansoko Sud is listed in Table 16.57.

Table 16.57 Kansoko Sud Mine Mobile Equipment List

Description	Maximum Number Required	Number of Units to Purchase	Number of Rebuilds
Drill Rig 282	8	13	11
LHD – 21 t	7	13	11
Haul Truck – 63 t	5	8	6
Concrete / Shotcrete Mixer Truck	4	4	N/A
Shotcrete Sprayer	4	6	N/A
Scissor Lift	5	14	N/A
Charmec – Explosives Loading Truck	4	14	N/A
Mine Support Equipment			
Explosives Transport Truck	4	15	N/A
Agicar – Concrete Mixing Truck – 12 m^3	2	2	N/A
Shotcrete – Backfill	2	5	N/A
Scissor Lift – Backfill	4	11	N/A
LVs	19	52	N/A
Personnel Carriers	5	14	N/A
Grader	1	5	N/A
Utility Equipment – Material	3	12	N/A
Utility Equipment – Maintenance	3	11	N/A
Telehandlers	4	15	N/A
Underground Mobile Crane	1	4	N/A
Skidsteer	3	4	N/A

16.6.9.2 Fixed Equipment

As described under Kakula, Section 16.3.9.2.

16.7 Kamoa 2 Underground Mining

16.7.1 Introduction

Based on updated design criteria, the mining method, mine design, and production schedule have been updated from previous studies. Mining method selection focussed 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 selected and the resultant mine designs and schedules.

The mining method used for the Kamoa 2 deposit is drift-and-fill. The Kamoa 2 Mineral Reserve by mining method is summarised in Table 16.58.

Table 16.58 Kamoa 2 Mine Probable Mineral Reserves by Mining Method

Production by Mining Method	Ore (Mt)	TCu (%)	Fe (%)	As (%)	S (%)	AsCu (%)
Ore Development	0.34	3.02	5.29	0.00	1.61	0.26
Drift-and-Fill	85.50	3.05	5.43	0.00	1.75	0.28
Total Ore*	85.84	3.05	5.43	0.00	1.75	0.28

*May not sum to total due to rounding.

16.7.2 Mine Design Parameters

16.7.2.1 Ore and Waste Properties

The Kamoa 2 deposit is a large stratiform copper deposit, typical of sediment hosted deposits. The deposit is tabular, with dips varying from 0–35°, with an average dip of 17°. The thicknesses varies from 3 m to 6 m. The ore zone density has been defined as using a greater than 2.4% Cu (total copper grade) cut-off. The swell factor for development is 50%.

Table 16.59 details the bulk density parameters of the ore and surrounding waste rock of the Kamoa 2 deposit.

Table 16.59 Bulk Density/In Situ by Area

Bulk Density/In Situ	Min (t/m^3)	Max (t/m^3)	Average (t/m^3)
Ore	2.27	3.23	2.81
Hanging wall	2.39	3.18	2.80
Footwall	2.21	3.04	2.67

16.7.3 Mine Planning

16.7.3.1 Lateral Development

The south and western declines have a maximum gradient of ±8.5°. The conveyor drift is 7.5 m W x 6.0 m H with 1.5 m arch corners, and 6 m W x 6.0 m H with 1.5 m arch corners for the service drift. 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 6 m W x 6 m H, with 1.5 m arch corners unless otherwise specified.

Conveyor drifts have a maximum gradient of ±9°. They are 7.5 m W x 6 m H (1.5 m arch corners), with re-mucks located every 300 m.

All perimeter drifts are 6 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 crosscuts 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° and 1.5 m arch corners.

16.7.3.2 Vertical Development

Vertical development consists of ventilation raises, bins, and boreholes. All ventilation raises are excavated with a raisebore drill. All ventilation shafts are designed at 6 m in diameter. Access drifts to the ventilation shafts are 6 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.

16.7.3.3 Drift-and-fill

The primary mining method for the Kamoa-Kakula deposits (drift-and-fill) was selected based on the results of a trade-off study which compared six different variations of drift-and-fill mining to identify the most suitable. The selected drift-and-fill mining method is explained in detail in Section 16.2.2.

A typical mining block with production drifts perpendicular to the connection drift is 198 m wide. Each mining block consist of three mining units. Each mining unit comprises three primary headings, three secondary headings, and three tertiary headings.

The production drift cross-sectional shape differs with the type of production heading. Typical primary and secondary production drifts have an 8 m wide arch (1.5 m arch corners). Slipe drifts are a 6 m wide arch (1.5 m arch corners).

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 (9°).

16.7.4 Backfill

Paste fill will be the primary backfilling strategy for the Kamoa 2 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 southern perimeter declines. Distribution pipes installed in the connection drifts will then deliver the paste fill to the production areas.

16.7.5 Mine Access Designs

Box-Cuts

There is one box-cut developed for access to the underground workings. The western box-cut incorporates two portals. Access to Kamoa 2 from the south is via the previously established Kamoa 1 box-cut and portal.

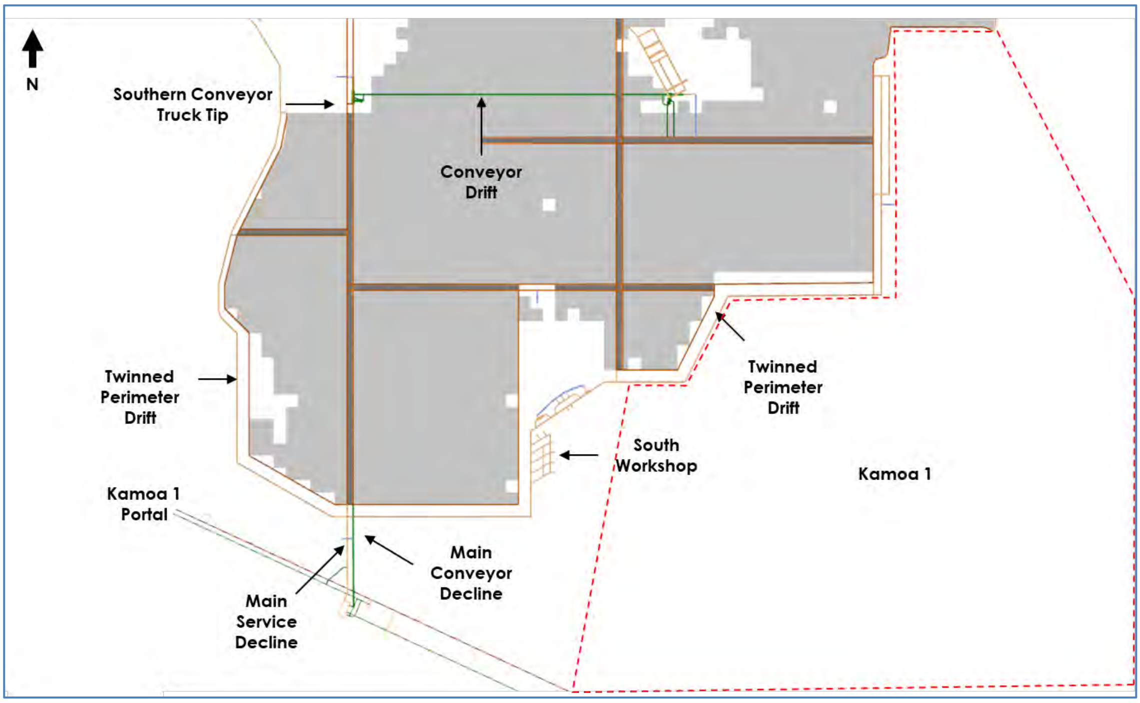
Main Declines

The deposit is accessed via twinned declines on the south, and west side, respectively.

One of the south declines is the primary mine service access, and the other decline is a conveyor haulage drift. Both the service access and conveyor drift join onto the previously established Kamoa 1 twinned decline to surface. The service decline has dimensions of 6.0 m W x 6.0 m H, with the conveyor decline 7.5 m W x 6.0 m H. Both southern declines have a maximum gradient of ±8.5° gradient.

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. Travel direction in the declines is uphill in the main conveyor decline and downhill in the main service decline. Approximately 325 m from the join to the Kamoa 1 decline (or 1,010 m from surface), 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 shown in Figure 16.82.

Figure 16.82 Kamoa 2 Mine Main South Access Development

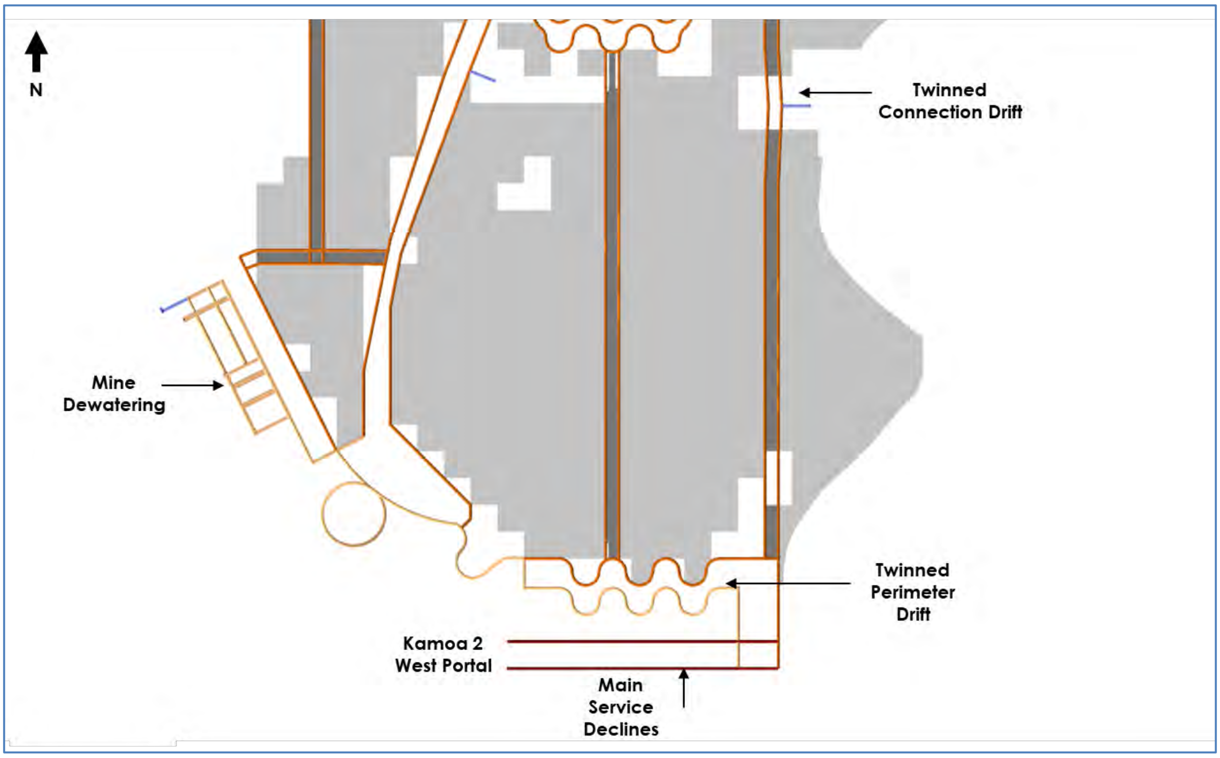


OreWin, 2023.

As haulage of material from the western side of Kamoa 2 will not utilise a conveyor system, the western access contains twin service access declines. The service declines have dimensions of 6.0 m w x 6.0 m H. Both western declines have a maximum gradient of ±8.5° gradient.

The western declines utilise one way traffic flow in order to minimise load and haul congestion. Approximately 511 m from the portal, twin access drifts are developed off the main service declines to the perimeter declines as shown in Figure 16.83.

Figure 16.83 Kamoa 2 Mine West Access Development



OreWin, 2023.

Perimeter Service and Conveyor Drifts

From the bottom of the west and south declines, 6 m w x 6.0 m H perimeter service drifts will be driven to the north-west, and north-east 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 conveyor drifts service the eastern side of the Kamoa 2 deposit. The east conveyor system extends to the lower eastern extremity of the orebody and converges onto the southern main conveyor decline. The main south decline rock handling system then transports ore towards the previously established Kamoa 1 conveyor decline for transport to surface. Conveyor drifts are 7.5 m w x 6.0 m H and are driven at a maximum gradient of ±9.0°. Section 16.7.8.4 covers the rock handling systems in greater detail.

Mining Areas

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

16.7.6 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. The Kamoa 2 production schedule will aim to provide material to the Kamoa Processing Plant as required.

Table 16.60 summarises the LOM development and production results.

Table 16.60 Kamoa 2 Mine Life-of-Mine Development and Production Summary

Waste Development	
Lateral (m)	88,593
Lateral (kt)	8,401
Vertical (m)	3,281
Vertical (kt)	216
Production by Mining Method	
Ore Development (m)	3,639
Ore Development (kt)	339
Drift-and-Fill (kt)	85,500
Total Ore Production	
Total Ore Development (kt)	339
Total Production (kt)	85,500
Total Tonnes (kt)	85,839
Diluted Grade	
TCu (%)	3.05
S (%)	1.75
As (%)	0.0011
Fe (%)	5.43
AsCu (%)	0.28

1. Notes: Vertical development includes boreholes.
2. Stope shapes designed on an NSR cut-off value of US$100/t NSR.

The following conditions were used in developing the LOM schedule:

- Proximity to the main accesses and initial 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.

16.7.6.1 Productivity Rates

Effective Operating Hours

The effective operating hours per shift are summarised in Table 16.61 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 16.61 Kamoa 2 Mine Shift Rotations and Effective Operating Hours Calculations

Shift Cycle	Calculations
Days per Year	360 days
Number of Crews in Rotation	3
Shifts per Day	2
Shift Duration	12 h
Travelling Time – In	19.50 min / 0.32 h
Travelling 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, and external consultants) and compared with inputs. The cycles were updated accordingly following team discussions. Mine productivities and schedule are based on a development rate of 120 m/month for all primary development.

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

The production rate in ore tonnes per month is highly variable as it is an outcome of the estimated development advance rate, combined with ore thickness, dilution parameters, and ore density.

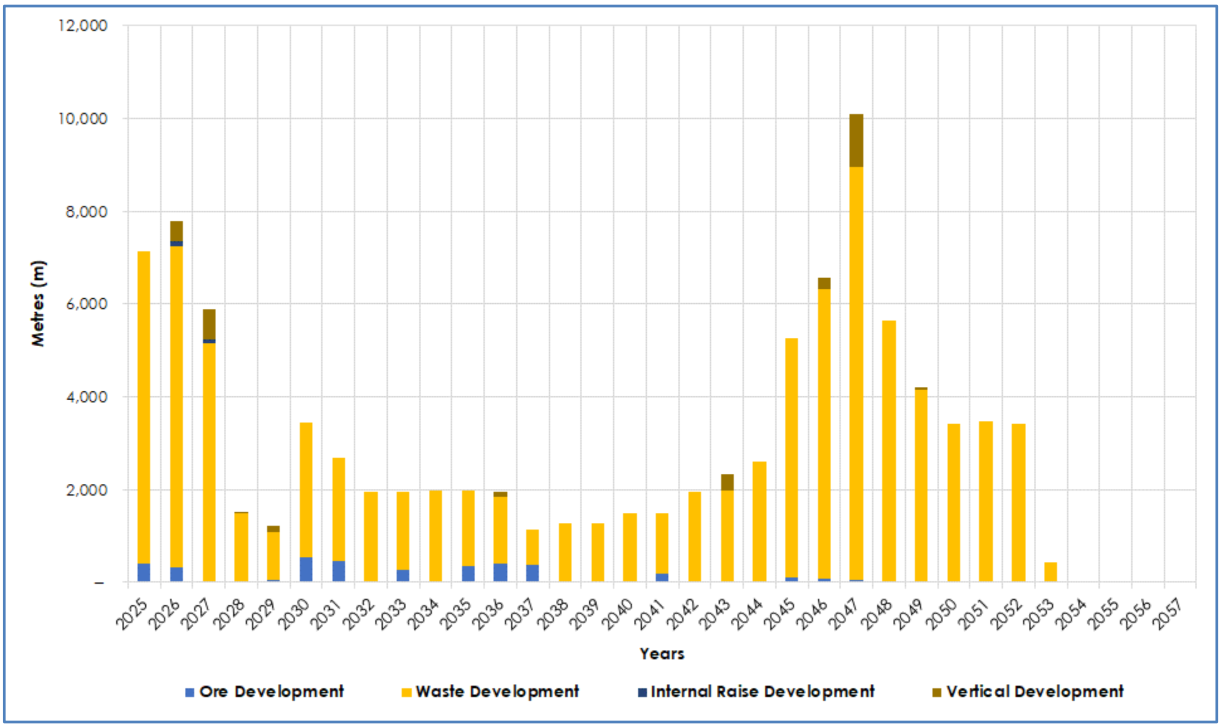
To determine the drift-and-fill production rates, development rates for primary, secondary, and tertiary drifts were combined with paste filling, cable bolting, and end-of-shift blasting restrictions in a block configuration to determine the net block production rate for use in the schedule. The net block production rate was then applied to the drift-and-fill mining shapes within the schedule. The production rate was adjusted depending on the height of the production heading in order to better represent the change in condition. Paste fill barricade construction/placement as well as installation of split sets (geotechnical supports) was not considered in the cycle calculations as it will be completed off critical task.

16.7.6.2 Development Schedules

Development schedule

The life-of-mine development schedule targets the areas required to support the LOM plan. This includes excavating the declines, perimeter service drifts, conveyor drifts, and key infrastructure associated with truck tips, ventilation, dewatering, and maintenance facilities in advance of production areas. Figure 16.84 illustrates the development metres associated with the LOM activities.

Figure 16.84 Kamoa 2 Mine Life-of-Mine Development Schedule



OreWin, 2023.

16.7.7 Mine Production Plan and Scheduling

As the Kamoa Phase 3 Processing Plant will be fed by material from Kamoa 1, Kansoko Sud, and Kamoa 2, a balance is formed between the three deposits. Kamoa 2 requires little initial development due to its proximity to the existing Kamoa 1 decline. Kamoa 2 will initially maintain a production rate of <0.5 Mtpa for five-years (2025–2030). The remainder of the Kamoa Phase 4 Processing Plant requirements (initially 5 Mtpa) will come from Kamoa 1, and Kansoko Sud. It is important to note that Kamoa 2 can increase production, however, the Kamoa Phase 3 Processing Plant does not require additional feed due to material from Kamoa 1 and Kansoko Sud. Table 16.62 presents the annual ramp-up scheduled tonnes for Kamoa 2.

Table 16.62 Kamoa 2 Mine Ramp-Up Scheduled Tonnage

Production Schedule	Years	Scheduled Tonnes (kt)
Ramp-Up (2025)	1	66
Ramp-Up Total	1	66

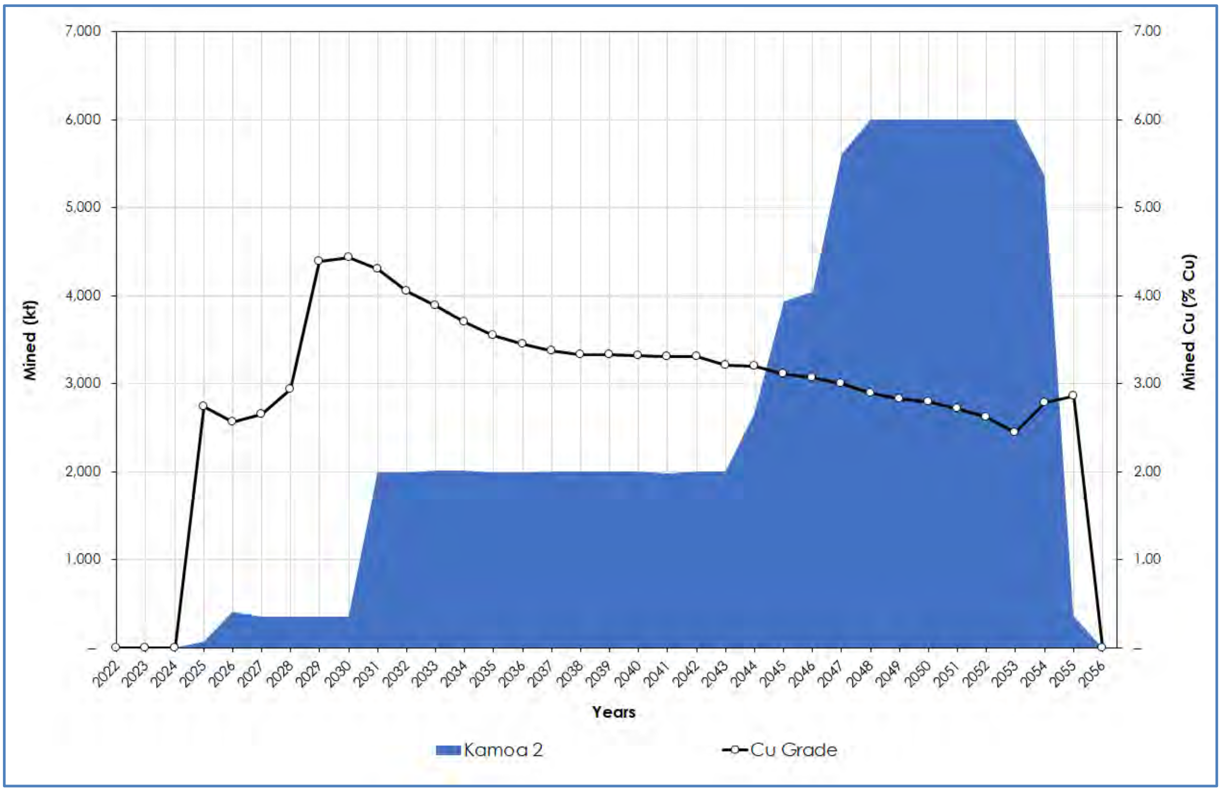
![OreWin logo]

Life-of-Mine Production Schedule

Initially, Kamoa Phase 3 Processing Plant will maintain a steady state of <0.5 Mtpa for five-years (2025–2030). As the Kamoa Phase 3 Processing Plant increases its rate from 5 Mtpa to 10 Mtpa, Kamoa 2 will ramp-up to a steady state production rate of 2 Mtpa to meet the milling requirement, with remaining material coming from Kamoa 1 (6 Mtpa), and Kansoko Sud (2 Mtpa). Kamoa 2 will maintain a steady state production rate of 2 Mtpa (2031–2043) until the Kansoko Sud deposit begins to taper off due to depletion. Kamoa 2 will then ramp-up and maintain its full production rate of 6 Mtpa to meet the Kamoa Phase 4 Processing Plant requirements, with the remainder coming from Kamoa 1. It is important to note that the Kamoa 2 production rate depends on Kamoa 1 and Kansoko Sud, as Kamoa 2 provides the Kamoa Phase 4 Processing Plant with the remainder to meets its 10 Mtpa processing target.

The mining blocks are scheduled so that a higher TCu value is achieved early in the mine life. Figure 16.85 illustrates the LOM production schedule and copper grade.

Figure 16.85 Kamoa 2 Mine Life-of-Mine Production Schedule and Copper Grade



OreWin, 2023.

16.7.8 Underground Infrastructure

16.7.8.1 Mine Ventilation System – Kamoa 2

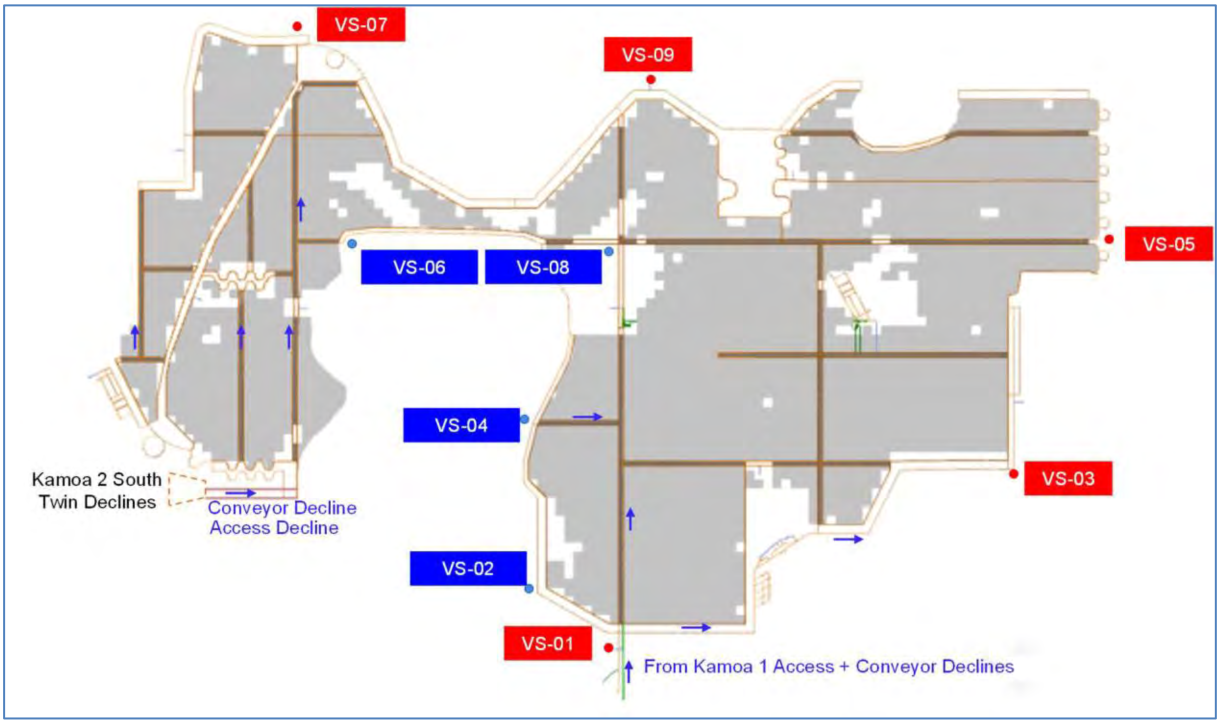
The underground mobile equipment fleet is diesel powered, and mine air cooling is required to maintain underground working air quality within the appropriate limits.

The following assumptions were considered in the ventilation design to maintain safe operating conditions underground and to abide by applicable legislative requirements. Australian and 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.
- Primary and secondary leakage rates used for preliminary airflow estimates for when the mine is fully developed are 10%, and 20% respectively. These factors are used 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 30 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.
- Heat load factors for diesel equipment have been split into four types. The following types of trucks, loaders, auxiliary and supporting equipment have respective heat load factors of 1.1, 1.0, 0.8, and 0.3, applied as linear activity tracks in the Ventsim modelling.
- Diesel engines are assumed to have a power conversion efficiency of 35%.

The mine layout schematic illustrating the location of the ventilation shafts is shown in Figure 16.86. The mine will be supplied with fresh air from four intake ventilation shafts located central to the orebody. The ventilation air will naturally flow through the perimeter drives to the extremities of the orebody. The ventilation will be extracted via four exhaust shafts located at the extremities of the orebody. VS-01 will ventilate the conveyor belt directly to return. 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 16.86 Kamoa 2 Mine Layout with Ventilation Shaft Locations



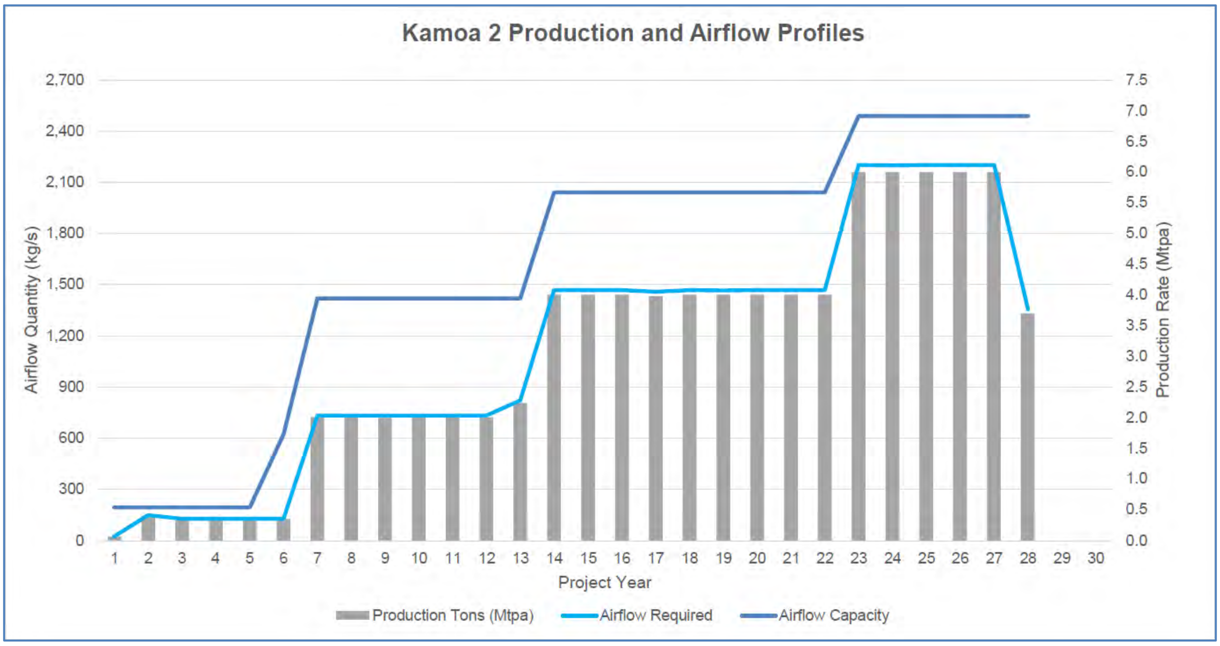
OreWin, 2023.

A summary of the primary ventilation fans is provided in Table 16.63. The LOM airflow requirements are shown in Figure 16.87.

Table 16.63 Kamoa 2 Mine Primary Main Ventilation Fan Requirements

Raise Location	No. of Fans (in parallel)	Operating Range (kg/s)	Peak Airflow (kg/s)	Peak Total Pressure at Collar) (kPa)	Estimated Power (kW)
VS-01	2	220 - 440	175	0.7 – 1.1	220 - 700
VS-03	3	660 - 750	506	1.1 – 1.5	1,040 – 1,610
VS-05	3	660 - 750	506	1.1 – 1.5	1,040 – 1,610
VS-07	3	660 - 750	506	1.1 – 1.5	1,040 – 1,610
VS-09	3	660 - 750	506	1.1 – 1.5	1,040 – 1,610

Figure 16.87 Kamoa 2 Mine Life-of-Mine Airflow Requirements



OreWin, 2023.

The model shows the primary ventilation requirements of 2,200 kg/s at the peak production rate of 6 Mtpa and a maximum depth of 360 m.

16.7.8.2 Mine Air Cooling Facilities – Kamoa 2

The cooling requirements were estimated from first principles and confirmed by VentSim models which considered auto-compression, surrounding and broken rock heat, vehicle heat, and all other heat components. The model indicated that no mechanical air cooling will be required for Kamoa 2 as fresh air from surface can ventilate the mine adequately at the required depth of 360 m below surface.

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 16.64.

Table 16.64 Kamoa 2 Mine Auxiliary Ventilation Fan Requirements

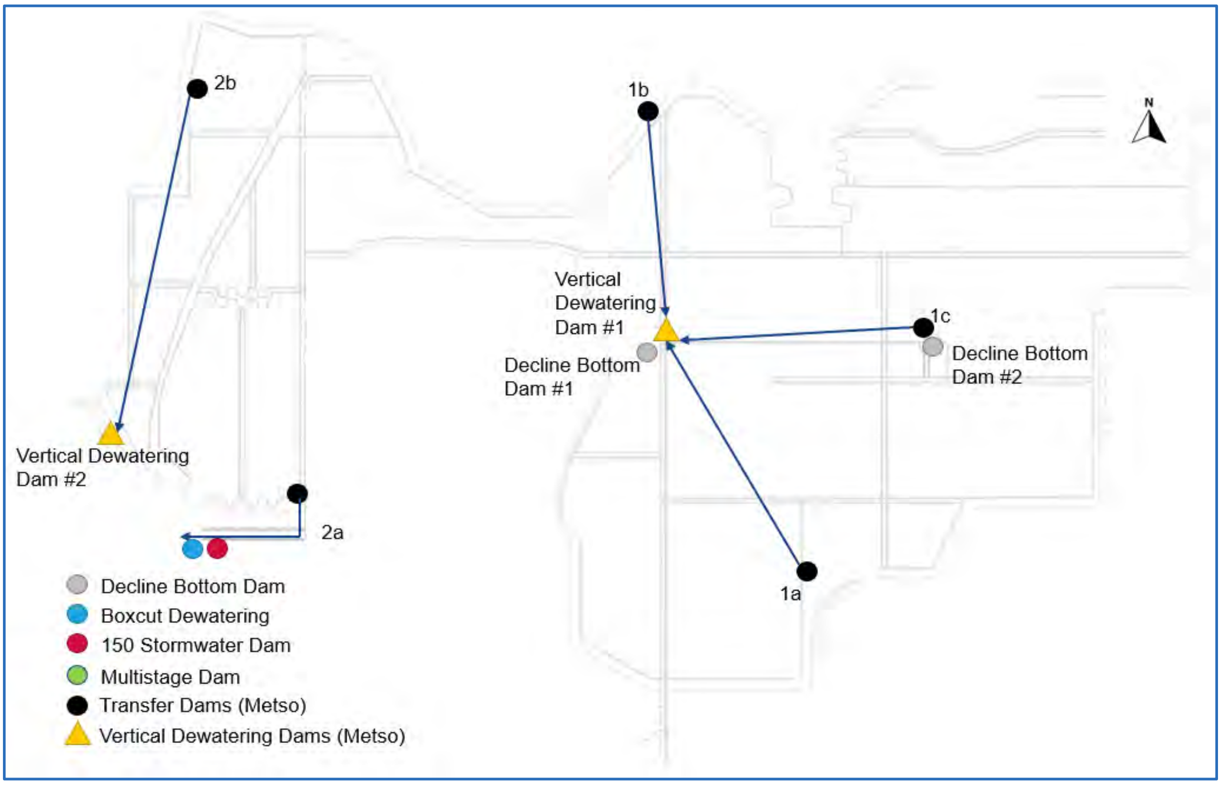
Location	Flow per Fan (m^3/s)	Fan Total Pressure (Pa)	Fan Size diameter (m)	Duct Type	Estimated Power (kW)
Development Headings	36	2,300	1.40	Flexible	110
Drift-and-fill Headings	36	2,300	1.40	Rigid and Flexible	110

16.7.8.3 Mine Dewatering

The Kamoa 2 orebody will be developed after Kamoa 1, and it is expected that Kamoa 2 will be largely dewatered by the time mining activity takes place. Kamoa #2 will have an expected peak fissure water inflow rate of approximately 560 l/s.

Two centrally located vertical dewatering pump stations will be required as indicated in Figure 16.88. Each dewatering pump station will be fed by transfer dam pump stations. The transfer dam pump stations, indicated as black dots in Figure 16.88, will have the capacity to pump 200–1,000 l/s to the central dewatering dams. The transfer dams will be strategically positioned to cover the required footprint of the mined ore body.

Figure 16.88 Kamoa 2 Mine Dewatering Layout

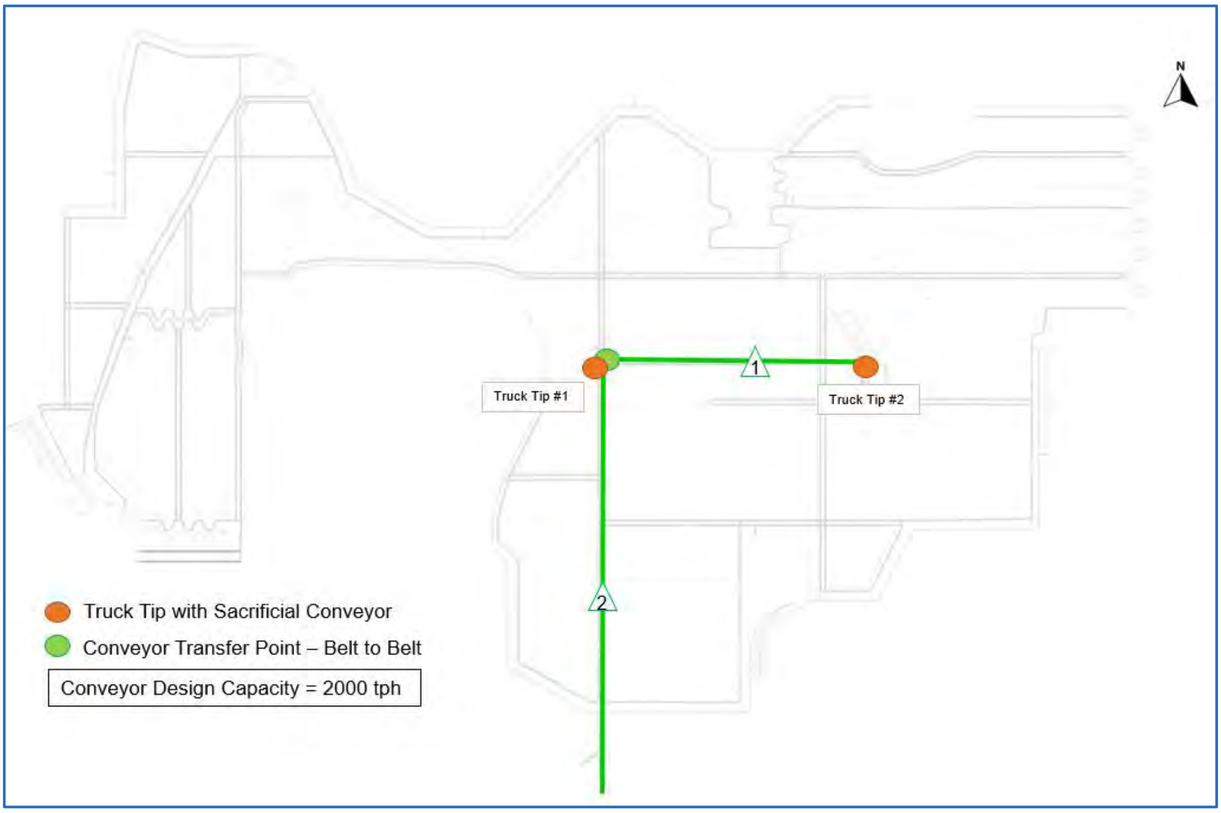


DRA, 2022.

16.7.8.4 Rock Handling

The underground rock handling system at Kamoa 2 consists of a South conveyor and an East Feed Conveyor which feeds onto the South conveyor. The South Conveyor delivers rock to the transfer silo at the Kamoa 1 main decline intersection. Figure 16.89 highlights the underground conveyor routing in Kamoa 2.

Figure 16.89 Kamoa 2 Mine Underground Rock Handling Conveyor Routing



DRA, 2022.

16.7.8.5 Materials Handling Logistics

The material handling procedures for Kamoa 2 will be the same as those for Kakula. Men and materials will be transported underground via the service decline shared with Kamoa 1, while emulsion, shotcrete, fuel, and lube will be transferred underground using boreholes. These facilities will be established in the vicinity of the main underground workshops.

16.7.8.6 Workshops

Kamoa 2 will have one centrally located main workshop and four satellite workshops on the strategically positioned throughout the mining area. Figure 16.90 below indicates the approximate positions for the main and satellite workshops.

Figure 16.90 Kamoa 2 Mine Workshops

DRA, 2022.

16.7.8.7 Fuel and Lubricant Distribution

Kamoa 2 will have two surface and two underground workshops. The two surface workshops each will require one 70 m³ diesel tank and four oil storage tanks. Two underground workshops are planned, and each will require the normal storage of three15 m³ diesel storage tanks and four oil storage tanks. One 4.5 m³ batch tank and three 2.5 m³ batch tanks for each of the underground workshops will be supplied for the storage and supply of hydrocarbons to the underground workings.

The total number of tanks planned for Kamoa 2 will be thirty-three.

16.7.8.8 Electrical Power Requirements

The electrical power requirements for all areas at Kamoa 2 are indicated in Figure 16.91 and Table 16.65 below.

Figure 16.91 Kamoa 2 Mine Life-of-Mine Maximum Demand Profile

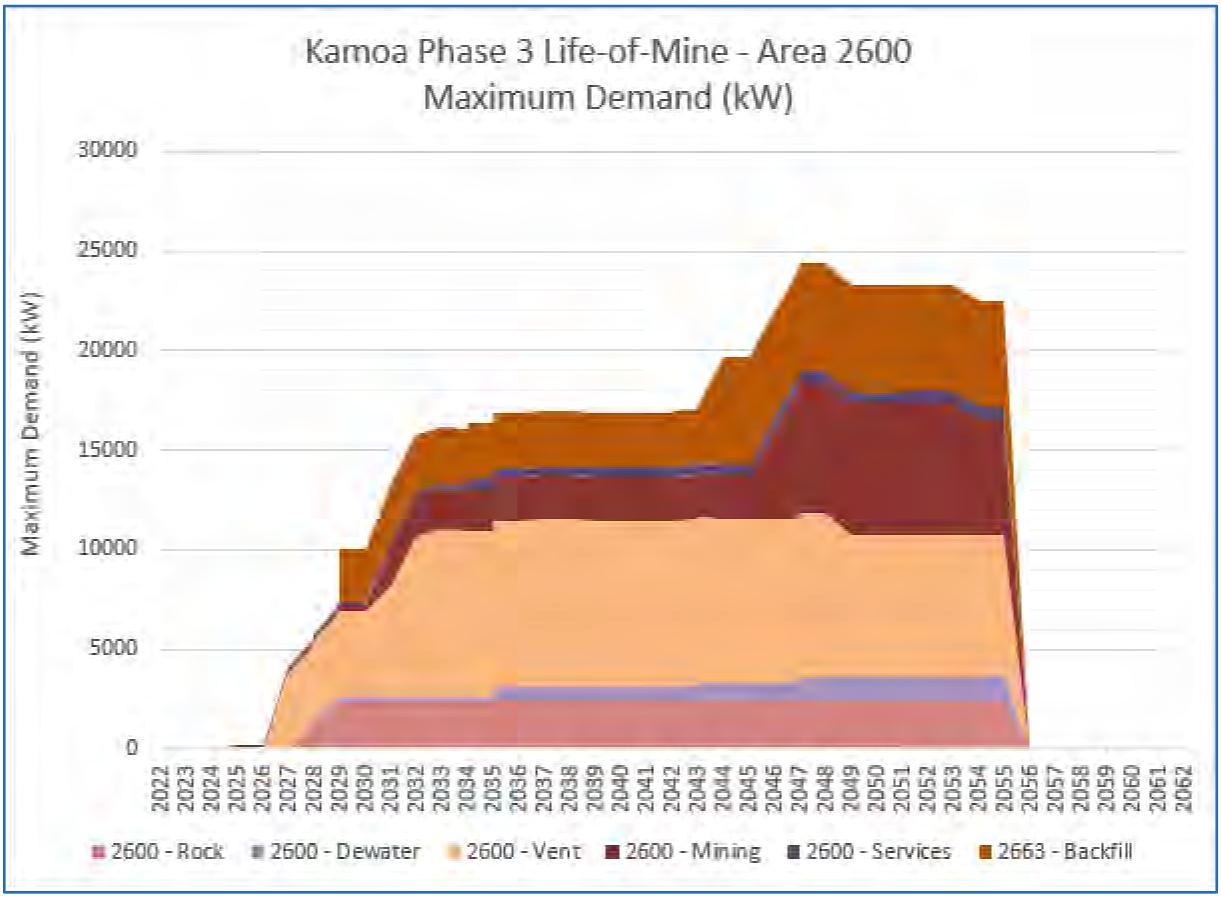


Table 16.65 Kamoa 2 Mine Life-of-Mine Maximum Demand

WBS - Area	Maximum Demand (kW) (2046)
2600 - Rock	2,385
2600 - Dewater	900
2600 - Vent	8,286
2600 - Mining	4,400
2600 - Services	491
2663 - Backfill	5,400
Total	21,863

16.7.8.9 Power Distribution

Kamoa #2 will be fed via the Kamoa #1 portal initially, and later when the box-cut is established, a Kamoa #2 portal substation will be established that will be fed from the Kamoa #1 33 kV infrastructure.

Underground power distribution will be via 11 kV cable distribution substations. For smaller loads, the mini substations will provide suitable LV supplies (690 V, 400 V, 1,000 V), as required in the area.

The 33 kV distribution substations, and 11 kV distribution substations allowed for are listed below. Power is distributed from these substations to various equipment substations.

- K2M 33 kV Portal Substation
- K2M – 33 kV VS-03 Substation
- K2M – 33 kV VS-04 Substation
- K2M – 33 kV VS-05 Substation
- K2M – 33 kV VS-07 Substation
- K2M – 11 kV Substation #1, 2873-SUB-001
- K2M – 11 kV Substation #2, 2873-SUB-002
- K2M – 11 kV Substation #3, 2873-SUB-003

16.7.9 Equipment

All equipment is sized for a 6.0 Mtpa case to support a drift-and-fill mining method. 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 based on the existing fleet and cover the major components for the operation.

16.7.9.1 Mobile Equipment

The mobile equipment is diesel-powered, rubber-tyred. 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.

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 for Kamoa 2 is listed in Table 16.66.

Table 16.66 Kamoa 2 Mine Mobile Equipment List

Description	Maximum Number Required	Number of Units to Purchase	Number of Rebuilds
Drill Rig 282	22	27	25
LHD – 21 t	19	27	23
Haul Truck – 63 t	15	17	17
Concrete / Shotcrete Mixer Truck	10	10	N/A
Shotcrete Sprayer	10	11	N/A
Scissor Lift	15	29	N/A
Charmec – Explosives Loading Truck	10	28	N/A
Mine Support Equipment			
Explosives Transport Truck	10	30	N/A
Agicar – Concrete Mixing Truck – 12 m^3	4	4	N/A
Shotcrete – Backfill	4	8	N/A
Scissor Lift – Backfill	10	21	N/A
LVs	55	112	N/A
Personnel Carriers	13	28	N/A
Grader	3	8	N/A
Utility Equipment – Material	9	25	N/A
Utility Equipment – Maintenance	7	22	N/A
Telehandlers	10	26	N/A
Underground Mobile Crane	3	7	N/A
Skidsteer	7	7	N/A

16.7.9.2 Fixed Equipment

As described under Kakula, Section 16.3.9.2.

17 RECOVERY METHODS

17.1 Introduction

The Kamoa-Kakula 2023 PFS considers an increase in production capacity from 7.6 Mtpa up to a total of 19.2 Mtpa achieved by debottlenecking of Kakula Phase 1 and Phase 2 to 9.2 Mtpa, and the phased addition of a further 10.0 Mtpa processing facility located at Kamoa.

The processing capacity of each module of the existing Kakula concentrator will be increased from 3.8 Mtpa to 4.6 Mtpa, through several modifications and hydraulic upgrades, to increase the Kakula complex processing capacity to 9.2 Mtpa. The new Kamoa concentrator will be constructed in a phased approach with two 5.0 Mtpa modules as the mining operations ramp-up to full production of 10.0 Mtpa. A phased approach further allows for increased processing flexibility and plant redundancy, while also reducing the peak capital demand by phasing of capital expenditure.

This section details the process and engineering design basis of the Kamoa Concentrator Plant and the Kakula Debottlenecking Project. The Kakula and Kamoa 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).

17.2 Kakula Concentrator Plant

17.2.1 Kakula Concentrator Basis of Design

The Kakula Debottlenecking Project is aimed at increasing current production at the Kakula complex to 9.2 Mtpa, by increasing processing capacity of each operating module to 4.6 Mtpa. The upgraded Kakula concentrator design criteria are shown in Table 17.1.

Table 17.1 Debottlenecked Kakula Concentrator Plant Design Criteria

Design Parameter	Unit	Design Value
Annual Surface Crushing Circuit Feed	Mtpa	9.2
Surface Crushing Circuit Availability	%	74
Surface Crushing Circuit Operating Time	hpa	6,482
Surface Crushing Circuit Feed Rate	t/h	1,430
Annual Milling Circuit Feed	Mtpa	9.2
Overall Milling Circuit Availability	%	91.3
Milling Circuit Operating Time	hpa	7,998
Milling Circuit Feed Rate	t/h	1,159
Milling Module Feed Rate	t/h/module	580
ROM Cu Grade	% Cu	4.43
Final Concentrate Grade	% Cu	50.0
Mass Pull to Final Concentrate	% Mill Feed	7.6
Cu Recovery	%	86.3

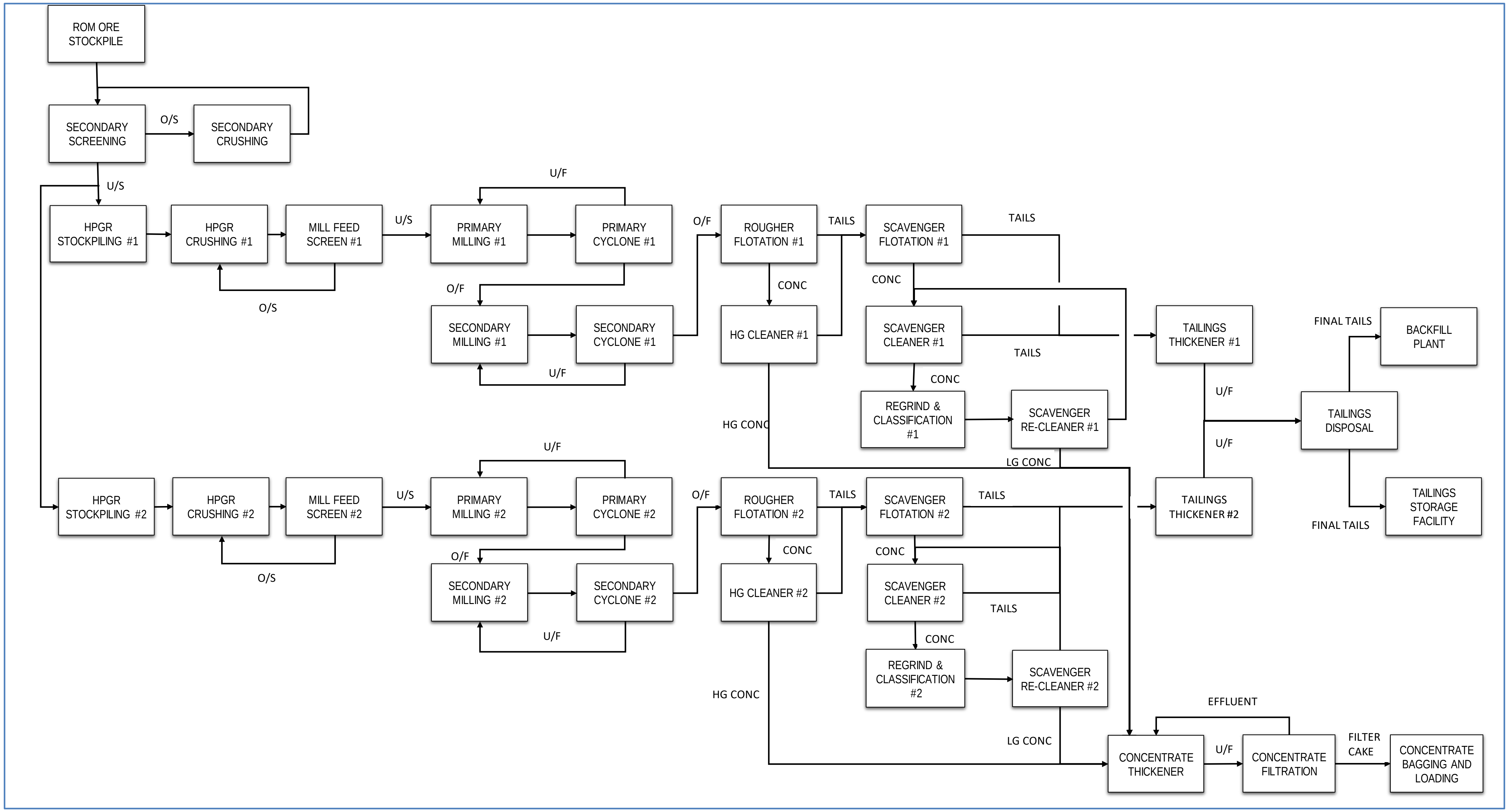
17.2.2 Plant Design and Process Description

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 shown in Figure 17.1.

Figure 17.1 Kakula Block Flow Diagram



DRA, 2020.

The following modifications are being made to increase the processing rate at Kakula Concentrator to 9.2 Mtpa (580 t/h fresh feed to milling per module):

17.2.2.1 Milling Circuit

The milling circuit upgrades and modifications are related to increasing hydraulic capacity to match the new processing volumes required. The main equipment upgrades in the milling circuit include:

- Upgrading of primary and secondary mill classification cyclone units to achieve target cut points.
- Upgrading of primary mill discharge pump motors to 900 kW.

17.2.2.2 Flotation Circuit

The flotation circuit upgrades and modifications are related to increasing hydraulic capacity and flotation residence time (specifically to the scavenger cleaner circuit) to match the new processing volumes required, without sacrificing on recovery. The main equipment upgrades in the flotation section include:

- Additional scavenger cleaning flotation capacity, complete with dedicated concentrate and tailings collection sumps and associated pumping systems.
- Installation of velocity breaker boxes at the high-grade cleaner feed trash screen feed.
- Upgrading of rougher flotation low-grade concentrate pumps motors.
- Replacing of larger aperture cloths on the high-grade cleaner feed trash removal screens.
- Upgrading of the high-grade cleaner tailings and concentrate pumps.
- Upgrading of the scavenger cleaner flotation tailings pump motors.
- Upgrading of the scavenger recleaner concentrate and tailings pumps.

17.2.2.3 Concentrate Regrind Circuit

A new regrind circuit feed surge tank and pumping system is provided as part of the debottlenecking project. In addition, the following modifications are required to increase the hydraulic capacity required to process 9.2 Mtpa:

- Replacing of the current regrind circuit classification cyclone clusters to achieve the target cut-points at the higher throughputs.
- Replacing of the regrind circuit product pumps with new units to match the increased flowrates.

17.2.2.4 Thickening Circuit

Due to an increase in design mass pull, and the combined effect of the additional processing capacity requirement, a second concentrate thickening circuit is required as part of the debottlenecking project.

Provision is made for a 3 m^2 concentrate thickener feed trash removal screen, which will remove any trash/foreign objects/fibres from the final concentrate product prior to reporting to the thickener feedbox. The concentrate will be dewatered to a pulp containing roughly 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 a 96 m^3 concentrate thickener effluent collection tank from where it is reused as process water. Provision is made for a 2-in-1 sampling system to sample the final concentrate.

In addition to a new concentrate thickener, provision is made for a temporary concentrate storage pond which will be used to store any overflow concentrate produced.

The final tailings thickener capacity is sufficient to deal with the increase in throughput, and the modifications required in the tailings thickener area pertains to piping system upgrades.

17.2.2.5 Concentrate Filtration

A fourth Larox unit (PF 72/72 M60) is being installed as part of the debottlenecking project, to cater for the increase in concentrate production.

The filtration feed tank area is further expanded by the addition of a third, mechanically agitated, filtration feed tank (500 m^3), complete with four filter feed pumps (two running, two standby). Material from the third feed tank can report to both Larox Filter #1 and Larox Filter #4.

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.

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 bunkers below each filter. Space allowance is made for future reversible, filter cake discharge conveyors, which in turn will either transfer the filter cake to the concentrate loadout conveyor or stockpiles. Filter effluent report to the concentrate thickening circuit.

Auxiliary systems to the fourth filter press units include hydraulic pressing system, dedicated filtrate and manifold wash systems, pressing air compressor and air receiver, and a drying air compressor and air receiver.

17.2.2.6 Tailings Disposal

Modifications in the tailings disposal circuit is focussed on increasing hydraulic capacity, and the level of redundancy, for risk management. A second final tailings sump is provided, complete with two additional centrifugal pumping trains (each train consisting of four pumps in series).

The motors on the existing high-pressure gland service water pump system is being upgraded to achieve the higher flowrate requirement from the additional pumping systems.

17.2.2.7 Services and Reagents

Provision is made for upgrading of the flocculant and coagulant dosing pumps, as well as the installation of additional hosing water pumps dedicated to the water monitoring gun system located at the concentrate storage pond.

17.3 Kamoa Concentrator Plant

17.3.1 Introduction

This section details the process and engineering design basis of the Kamoa Concentrator Plant. The Kamoa 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.

17.3.2 Kamoa Concentrator Basis of Design

The Kamoa concentrator is designed to process a maximum throughput of 10.0 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 Kamoa concentrator design criteria are shown in Table 17.2.

Table 17.2 Kamoa Concentrator Plant Design Criteria

Design Parameter	Unit	Design Value
Annual Surface Crushing Circuit Feed	Mtpa	10.0
Surface Crushing Circuit Availability	%	72
Surface Crushing Circuit Operating Time	hpa	6,307
Surface Crushing Circuit Feed Rate	t/h	1,585
Annual Milling Circuit Feed	Mtpa	10.0
Overall Milling Circuit Availability	%	91
Milling Circuit Operating Time	hpa	7,972
Milling Circuit Feed Rate	t/h	1,254
Milling Module Feed Rate	t/h/module	627
ROM Cu Grade	% Cu	3.49
Final Concentrate Grade	% Cu	37.0
Mass Pull to Final Concentrate	% Mill Feed	8.2
Cu Recovery	%	87.0

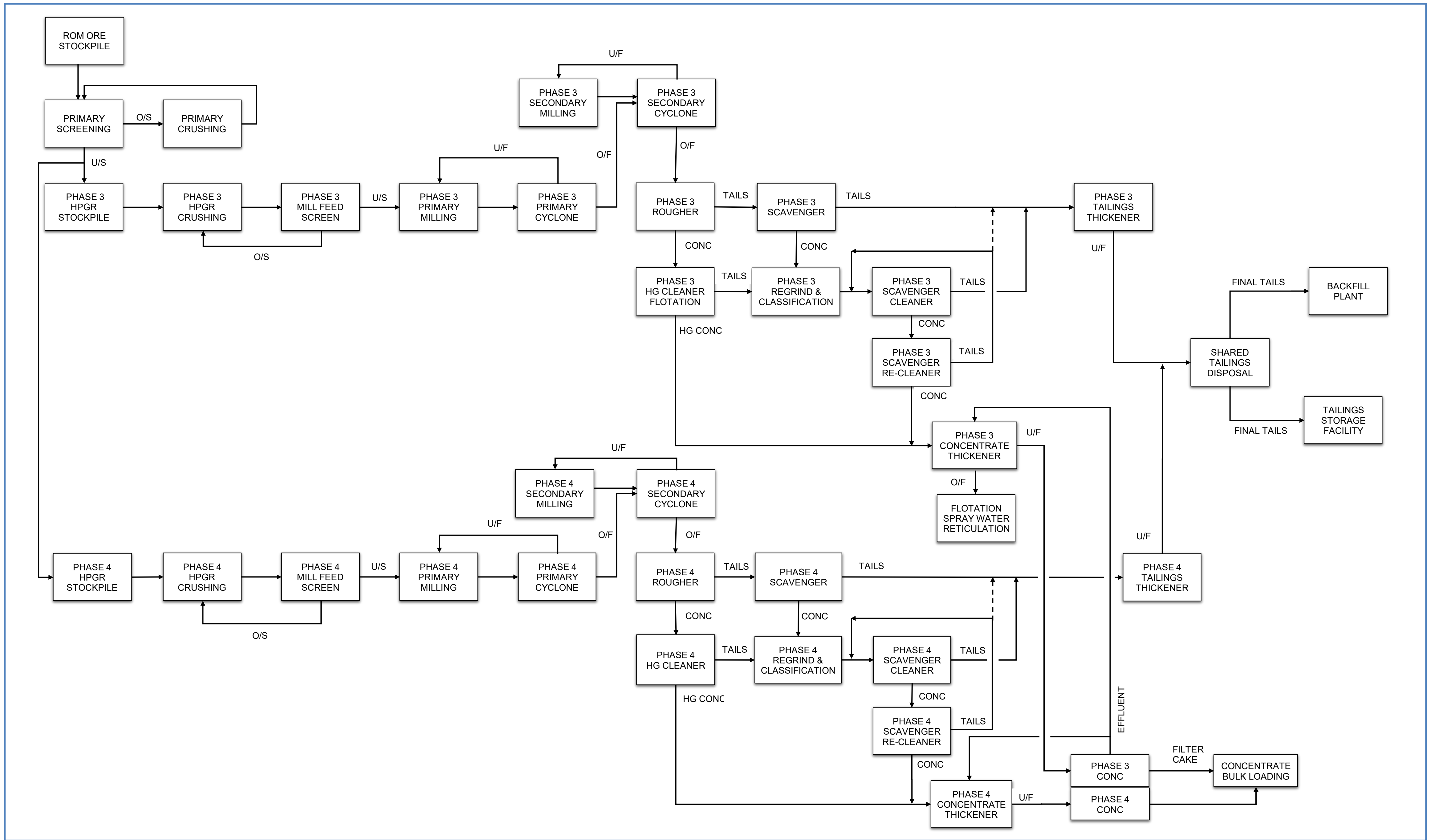
17.3.3 Plant Design and Process Description

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.
- Two identical concentrate thickening circuit
- Shared concentrate filtration circuit.
- Two identical tailings thickening circuits, with a shared tailings thickener overflow clarifier.
- Two identical tailings disposal and backfill plant feed systems.
- Shared utility systems for air, water and reagents.

A high-level block flow diagram of the Kamoa concentrator plant is shown in Figure 17.2.

Figure 17.2 Kamoa Block Flow Diagram



DRA, 2023.

A description of the main components of the process follows.

17.3.3.1 Run-of-Mine Reclamation

ROM ore with a lump size (F100) of 450 mm from underground, is conveyed to a single 20,000 t ROM stockpile for storage prior to the crushing circuit. The material is extracted from the stockpile, at a controlled rate via two variable speed apron feeders and is discharged onto the primary screening feed conveyor.

Provision is made for a dust control system in the area, as well as a process camera for monitoring.

17.3.3.2 Crushing and Screening

The primary screening feed conveyor transfers material from the ROM stockpile, together with primary crusher product, to the 300 t primary screening feed bin. The material is screened at 50 mm using two 3.6 x 7.0 m, dual deck, vibrating screens. The primary screen oversize material, roughly 60% of the screen feed, is conveyed to the primary crushing circuit for size reduction, while the primary screen undersize material reports to dedicated HPGR feed stockpiles via dedicated primary screen undersize transfer conveyors. The Phase 3 installation caters for a single screen, with the second unit to be installed as part of Phase 4.

The primary screen oversize material reports to the 350 t primary crushing feed bin, via the primary crushing feed conveyor. The material is extracted at a controlled rate using dedicated feeding systems to eventually feed four continuously operating cone crushers (Model: CS660). The Phase 3 installation caters for two cone crushers, with another two units to be installed as part of Phase 4. Each primary cone crusher is installed with a 315 kW motor to achieve a size reduction from F_{80} 195 mm to P_{80} 52 mm. The primary cone crusher product is conveyed to the primary screening feed bin via the primary screening feed conveyor.

Tramp iron removal systems are included on the primary screen feed conveyor and primary screen oversize transfer conveyor, with metal detection on the primary crusher feed conveyor.

Provision is made for dust suppression at the screening and crushing buildings. Process cameras are provided at the primary screening building for monitoring.

17.3.3.3 HPGR Stockpiling

The HPGR stockpiling installation are identical for Phase 3 and Phase 4. The primary screening undersize product from each screen is conveyed by dedicated conveyors to the two HPGR feed stockpiles.

Each HPGR feed stockpile is designed to store a live capacity of 7,500 t. The material is extracted from these stockpiles, at a controlled rate via two variable speed belt feeders per stockpile, 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.

17.3.3.4 HPGR Crushing

The HPGR crushing installation are identical for Phase 3 and Phase 4. The primary 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 primary screen undersize product is combined with the primary mill feed screen oversize recycle stream.

The combined HPGR feed material reports to the 260 t HPGR feed bin, via the HPGR feed bin conveyor. The material from the HPGR feed bin is extracted at a controlled rate using the HPGR feed conveyor. Tramp iron handling systems are included on the HPGR feed conveyor in the form of a metal detection unit as well as a bypass chute arrangement, that will activate and bypass the HPGR when the metal detector identified metal in the stream.

The HPGR unit, a Polycom HPGR 17/12-5, is installed with two 1,200 kW drives to achieve a size reduction from F100 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 3.6 m × 7.3 m multi-slope vibrating unit, is utilised for primary mill feed classification. The primary mill feed screen oversize product (+8 mm) is collected on the HPGR feed bin conveyor and recycled to the HPGR circuit for size reduction. The screen undersize material (–8 mm) gravitates to the primary mill feed hopper where it combines with the primary mill classification cyclone underflow.

17.3.3.5 Primary Milling

The primary milling installation are identical for Phase 3 and Phase 4. Each module comprises of a 23'Ø × 34' EGL, overflow discharge ball mill (installed with dual 5,000 kW VSDs) operating in closed circuit with a cyclone cluster consisting of 12 x 660 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 170 m^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 done via using bags, 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 bunkers. The design allows for one mill relining per phase. Provision is made for process cameras at the primary mill feed screen areas, for monitoring purposes.

17.3.3.6 Secondary Milling

The secondary milling installation are identical for Phase 3 and Phase 4. Each module comprises of 23'Ø × 34' EGL overflow discharge ball mills (installed with dual 5,000 kW VSDs), operating in closed circuit with a cluster of 22 x 350 mm diameter classification cyclones.

The primary milling classification cyclone overflow products report to the 170 m^3 secondary milling discharge sump in a reversed feed configuration, where it combines with the secondary mill product, prior to being fed to the secondary mill classification cyclone at a controlled rate and density.

The secondary mill classification cyclone underflow product gravitates to the secondary mill feed hopper, while the cyclone overflow product (P_{80} 53 µm) reports to the mechanical agitated, 500 m^3 rougher flotation surge tank via a two-stage metal accounting sampling installation and trash removal system. 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 done using a magnet and sputnik arrangement, to load the media to the secondary mill feed hopper. Dosing of reagents is 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.

17.3.3.7 Rougher/Scavenger Flotation

The rougher flotation circuit installation is identical for Phase 3 and Phase 4. The rougher flotation circuit consist of a single bank of eleven 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 – three cells and gravitates directly to the 75 m^3 high-grade cleaner feed sump. Provision is made for dosing of collector, promoter, and frother to the rougher high-grade cleaner feed sump, to allow for conditioning of the high-grade cleaner feed slurry.

A low-grade concentrate product is produced from the remaining cells and gravitates to the 22 m^3 low-grade rougher concentrate sump from where it is pumped to the concentrate regrind circuit using a fixed speed, duty / standby pumping system. Provision is made in the design to divert the third and fourth 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 first, and seventh, cell's feed box.

The scavenger tailings product gravitates to the 45 m^3 rougher tailings sump via a two-stage sampling system before being pumped to a dedicated tailings thickener using a variable speed, duty / standby pump system. Provision has been made to recycle the scavenger cleaner tailings to the scavenger flotation circuit for additional scavenging.

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.

17.3.3.8 High-Grade Cleaner Flotation

The high-grade cleaner flotation circuit installation is identical for Phase 3 and Phase 4. The high-grade cleaner flotation circuit consist of a single low entrainment Jameson flotation cell, to produce the final high-grade concentrate product.

The high-grade rougher concentrate gravitates to the high-grade cleaner feed sump, from where it is pumped to the Jameson flotation cell. The high-grade cleaner concentrate gravitates to the 15 m^3 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 tailing from the high-grade cleaner cell gravitates to the 20 m^3 high-grade cleaner tails sump from where it is pumped to the regrind milling circuit, at a controlled rate using a variable speed duty / standby pump system.

Spillage produced in the high-grade cleaner area is collected in a dedicated spillage sump and pumped to the high-grade cleaner tailing's sump via a vertical spindle pump.

17.3.3.9 Concentrate Regrind Milling

The concentrate regrind circuit installation are identical for Phase 3 and Phase 4. The concentrate regrind milling circuit comprises two high intensity 1,800 kW HIG regrind mills, each operating in open circuit with a cluster of 20 x 150 mm diameter cyclones.

The regrind mill feed consisting of the low-grade rougher / scavenger concentrate, together with the high-grade cleaner tailings (cyclone feed F_{80} 35 µm) is pumped to the 220 m^3, mechanically agitated, Concentrate Regrind Feed Tank, from where it is pumped at a controlled rate and density to the regrind mill 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 60 m^3 regrind mill product sump.

The cyclone underflow product (P_{80} 80 µm) of each cluster gravitates to a feed sump from where it is pumped 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 cleaner flotation cell using a variable speed, duty / standby pumping system. Online grade measurement is provided on the scavenger cleaner 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 cleaner feed trash screen for monitoring purposes.

17.3.3.10 Scavenger Cleaner Flotation

The scavenger cleaner flotation circuit installation is identical for Phase 3 and Phase 4. The scavenger cleaner flotation circuit comprises of a single bank of nine160 m³ mechanically agitated forced air flotation tank cells in series.

The scavenger cleaner feed has 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 medium, or low-grade concentrate product can be produced by the scavenger cleaner circuit, depending on feed grade to the circuit. Low-grade concentrate gravitates to the 18 m³ low-grade concentrate sump from it is pumped to the scavenger recleaner circuit via a variable speed, duty / standby pump system.

The low-grade concentrate gravitates to an 18 m³ sump from where it is pumped to the scavenger recleaner circuit via a fixed speed, duty / standby pump system.

The medium-grade concentrate gravitates to a 7 m³ sump from where it is pumped to the concentrate thickener circuit via a fixed speed, duty / standby pump system.

The scavenger cleaner tailings gravitate to the 45 m³ scavenger cleaner tailings sump via a two-stage sampling system from where it is pumped to the tailings thickener, to combine with the scavenger tailings product from the same module.

Scavenger cleaner area spillage gravitates to the spillage sump from where it is pumped back to the head of the scavenger cleaner flotation bank for cleaning, using a vertical spindle pump. Emergency showers are included in strategic areas.

17.3.3.11 Scavenger Recleaner Flotation

The scavenger recleaner flotation circuit installation is identical for Phase 3 and Phase 4. The scavenger recleaner circuit consist of two low entrainment Jameson flotation cells.

The scavenger cleaner concentrate product is pumped to the first scavenger recleaner cell pump sump (feed box), where it is combined with the required collector, promoter and frother, prior to upgrading. The tailings from the first scavenger recleaner cell is pumped to the second recleaner cell for final upgrading.

The concentrate products from both scavenger recleaner cells gravitates to a common, 10 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 second scavenger recleaner cell gravitates to the 20 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.

17.3.3.12 Flotation Tailings Thickening

The tailings thickening installation is identical for Phase 3 and Phase 4. The flotation tailings is pumped to a 47 m diameter, high rate thickener unit.

The scavenger tailings together with the scavenger cleaner tailings report to the tailings' thickener feed box, where it is combined with flocculant before gravitating to the thickener feedwell. The thickener systems allow for automatic internal dilution systems.

The tailings are thickened to an underflow product targeting 58% 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 a common tailings clarifier, 50 m diameter unit. From here water overflows to dedicated 3,000 m^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.

17.3.3.13 Backfill Feed System and Final Tailings Disposal

The backfill and tailings transfer systems are identical for Phase 3 and Phase 4.

Thickened flotation tailings and clarifier underflow is pumped to a two-stage metal accounting sampling system before gravitating to the mechanically agitated, 240 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 mechanically agitated 100 m^3 final tailings tank 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 pump train has four high-pressure centrifugal pumps in series, delivering slurry to the TSF via dedicated pipelines. In future, a fifth pump needs to be added to each pump train when pumping to the Mupenda tailings facility.

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 gravitates to the spillage collection sump from where it is pumped to the final tailing's sump using a submersible pump.

17.3.3.14 Concentrate Thickening

The concentrate thickening systems are identical for Phase 3 and Phase 4. The flotation concentrate products are pumped to a 24 m diameter, high-rate thickener unit. The high-grade concentrate product reports to a dedicated 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.

17.3.3.15 Concentrate Filtration Feed

The thickened concentrate is pumped to a concentrate feed box, which can gravitate to either one of two 520 m^3 mechanically agitated filtration feed tanks, from where a total of three 156 m^2 horizontal plate pressure filters will be fed using a variable speed, duty / standby pump sets.

Spillage produced in the filtration feed area gravitates to a spillage collection sump from where it is pumped to the concentrate filtration feed tank splitter box using a submersible pump.

The above installation will be duplicated as part of Phase 4.

17.3.3.16 Concentrate Filtration

The thickened concentrate is dewatered to a filter cake at a target moisture of 10.0% solids (w/w), using three 156 m^2 horizontal plate pressure filters.

The filter cake product reports to dedicated bunkers below each filter. Space allowance is made for future reversible, filter cake discharge conveyors, which in turn will either transfer the filter cake to the concentrate loadout conveyor or stockpiles. Filter effluent report to the concentrate thickening circuit.

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.

The above installation will be duplicated as part of Phase 4.

17.3.3.17 Air Services

Flotation Blower Air

Low-pressure blower air to each of the forced air tank flotation cells are supplied by five fixed speed multistage centrifugal air blowers, of which three units will be installed in Phase 3. 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.

Compressed Air

The design includes four air compressors, supplied as vendor packages, dedicated to the concentrator plant. Only two units will be installed as part of Phase 3.

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.

Filtration Air

Drying air to the concentrate filter units is supplied by two 1,300 kPa compressors (drying pressure at 1,100 kPa) in a duty / standby configuration and stored in two 40 m^3 drying air receivers, 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 kPa) in a duty / standby configuration and stored in a single 6 m^3 pressing air receiver from where it is distributed to either one of the filter units.

The above systems will be duplicated for Phase 4.

17.3.3.18 Water Services

The water circuit design for the concentrator circuit consists of three separate systems, i.e. process water, filtered water and potable water.

Process Water

The final installation includes two 3,000 m^3 process water tanks of which one will be installed in Phase 3. These tanks are fed by any excess concentrate thickener effluent, TSF return water, and tailings thickener overflow products. Process water, required for dilution, is distributed to each concentrator module via a dedicated duty / standby process water pump system. Process water is further used for general flushing and hosing, supplied by a dedicated duty / standby flushing and hosing pump systems per module.

The design further includes a process water supply to the backfill plant.

Filtered Water

The sand filter plant is fed by a mixture of TSF return, backfill effluent, underground water, and raw water. A sand filter plant per module will be installed.

Filtered water product from the sand filter plants is pumped to the shared 1,500 m^3 filtered water tank (installed in Phase 3), 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 ring mains 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.

A second set of gland seal water and filtered water pumps will be installed in Phase 4.

Potable Water

Potable water is supplied from treated borehole water that is pumped to the Kamoa site potable water tank close to the Kamoa box-cut. From here it is distributed to the required points and used for human consumption, reagent mixing and safety showers.

The Phase 3 and Phase 4 circuits will share a fire water system, consisting of two 1,000 m^3 storage tanks and distribution pump system.

17.3.3.19 Collector Make-Up and Dosing

Sodium IsoButyl Xanthate (SIBX) is used as the main collecting reagent in the flotation circuit. SIBX is delivered in powder form (850 kg bags) and stored in the reagent store. As required, bags are moved from the reagent store to the SIBX make-up area.

During batch make-up, a bag is manually hoisted and discharged into the 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 here it is distributed to the 10 m^3 day tank using a duty / standby peristaltic pump system. From the day tank required volumes will be pumped to each dosing point using dedicated dosing pumps.

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.

During Phase 4, a second 10 m^3 day tank complete with dedicated dosing pumps per dosing point will be installed. Dedicated duty / standby transfer pumps from the dosing tank to the new day tank will be installed.

17.3.3.20 Promoter Make-Up and Dosing

AERO 3477 is used as promoter in the flotation circuit and is delivered as a 50% (w/v) liquid 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 m^3 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 m^3 promoter dosing tank using the same transfer pumps. From here it is distributed to the 5 m^3 day tank using a duty / standby peristaltic pump system. From the day tank required volumes will be pumped to each dosing point using dedicated dosing pumps.

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.

During Phase 4, a second 10 m^3 day tank complete with dedicated dosing pumps per dosing point will be installed. Dedicated duty / standby transfer pumps from the dosing tank to the new day tank will be installed.

17.3.3.21 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 transferring of the neat frother from these IBCs into a 5 m^3 frother day tank, from where it will be dosed, without any further dilution, using dedicated variable speed peristaltic pumps per dosing point.

No additional spillage handling systems are included in the design, as the IBCs are located within the flotation area bunds.

During Phase 4, a second 5 m^3 day tank complete with dedicated dosing pumps per dosing point will be installed.

17.3.3.22 Flocculant Make-Up and Dosing

The design allows for use of BASF Magnafloc 10 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 m^3 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 systems. 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.

Phase 4 will include dedicated duty / standby pump systems for each of the two additional thickener installations.

17.3.3.23 Coagulant Make-Up and Dosing

The design allows for coagulant addition at both tailings thickeners and the tailings thickener clarifier. 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 coagulant into either one of the two 100 m^3 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 dosing points using dedicated duty / standby pump systems. The coagulant is further diluted to 0.05% (w/v) at the dosing points using inline mixers and filtered water.

Spillage produced in the coagulant 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.

Phase 4 will include a dedicated duty / standby pump systems to dose to the scavenger tailings area.

17.3.4 Concentrator Services Requirements

Table 17.3 lists the estimated projected water, consumables and power requirements for the concentrator.

Table 17.3 Projected Concentrator Water, Power, and Consumables

Item	Description	Consumption per tonne of Plant Feed	Annual Requirement (Phase 3 & Phase 4)
Power	Electricity	58.4 kWh/t	584 GWh
Water	Raw make-up	0.20 m^3/t	2,400 ML
Reagents	Frother	203 g/t	2,088 t
	Collector	310 g/t	3,456 t
	Promotor	57 g/t	1,128 t
	Flocculant	35 g/t	352 t
	Coagulant	17 g/t	168 t
Consumables	Grinding media (70 mm steel balls)	0.350 kg/t	3,504 t
	Grinding media (30 mm steel balls)	0.450 kg/t	4,512 t
	Grinding media (3 mm Ceramic)	20 g/kWh	751 t

17.4 Kamoa-Kakula Copper Smelter

17.4.1 Introduction

This section details the process and engineering design basis of the Kamoa Smelter, which is expected to process about 1.2 million tonnes per annum (Mtpa) of concentrate to produce about 500 ktpa of cast copper anode. The process design is based on testwork findings and assessments included in Section 13, trade-off studies, prospective vendor package information and relevant design information from similar circuits. The process plant design was undertaken by China Nerin Engineering Co. Ltd ("Nerin").

The Direct to Blister Furnace (DBF) technology selected for the Kamoa Smelter has greater oxygen and fuel consumption efficiency compared to other technologies, which helps to reduce the overall operating costs of the smelter. It requires fewer major equipment items than some competing processes, enabling it to offer competitive capital costs. The main deficiency of DBF technology is the high percentage of copper that reports to slag. This is because a significant amount of copper reports to slag in the converting step due to the production of blister copper in the furnace, rather than copper matte. The DBF technology therefore requires the inclusion of a Slag Cleaning Furnace (SCF) and in this case the SCF slag will need to be processed further by flotation to recover residual copper for recycling to the smelter.

In the second phase of the Kamoa project, after exhaustion of the Kakula ore body, it may no longer be possible to produce blister copper in a single DBF furnace due to the lower copper and higher iron content of the concentrate from the Kamoa ore body. A more conventional arrangement of a Flash Smelting Furnace (FSF) followed by a Flash Converting Furnace (FCF) to produce blister copper has been proposed in the study, although it is understood that alternatives are still being considered.

The overall smelting process proposed for Kamoa-Kakula includes the following main steps:

- Concentrate Drying
  - Filter cake storage and blending.
  - Steam dryer with bag filter (baghouse) and dedicated exhaust stack.
- Flash Smelting and Converting
  - Concentrate, oxygen, diesel, coal, burnt lime and flue dust fed systems to furnace.
  - Direct to Blister Furnace (DBF).
  - DBF waste heat boiler and electrostatic precipitator.
  - Fugitive gas collection and bag filter.
  - Electric Slag Cleaning Furnace with evaporative cooler and dust cyclones.
  - Blister copper tapping and transfer to anode furnace.
- Anode Furnaces
  - DBF and SCF molten blister copper transferred to anode furnaces.
  - Rotary anode furnaces with evaporative coolers.
  - Anode copper to dual casting wheels, containing up to 99.7% copper.
  - Anode Furnace slag recycled to Slag Cleaning Furnace.
- Slag Cleaning Furnace
  - Copper metal to Anode Furnaces (Phase 1). In Phase 2 the SCF will become redundant.
  - SCF Slag to slow cooling and crushing area.
- Double Contact Sulfuric Acid Plant
- Acid Plant Tail Gas Desulfurisation
  - SCF Gas Scrubber followed by a mixing chamber for fugitive gases, acid plant tail gas and SCF gas.
  - Desulfurisation of combined gas stream with lime, producing gypsum for disposal.
  - Wet electrostatic mist precipitator before discharging gas to the main smelter exhaust stack.
  - Wastewater streams to a treatment plant.
- Slag Cleaning Furnace Slag Flotation Circuit
  - Slag slow cooling, crushing and milling.

- Slag flotation sulfidisation and reagent systems.
- Slag flotation.
- Slag flotation concentrate and tailings dewatering and handling.

Anode copper from the smelter will be sold, for refining by others.

17.4.2 Kamoa-Kakula Smelter Basis of Design

The standards adopted for the project are shown in Table 17.4 other than environmental standards which are expressed as specific waste stream concentrations for each major pollutant.

Table 17.4 Design Standards for the Kamoa-Kakula Smelter

Discipline	Standard
Electrical	IEC (Chinese standards if not mentioned in IEC)
Civil Works	Chinese standards
Structural	Chinese standards (Relevant standard for mechanical packages which including structures)
Architecture	Chinese standards
Instrumentation	ISA, IEC
Communication	Chinese standards
Mechanical	Relevant standards where the manufacturer is based
Piping	Chinese standards
Valves and Flanges	Chinese standards
Sanitation and Fire Protection	As per Kamoa standards
Safety and Environment	IFC

The nameplate design capacity for the smelter is 500 ktpa of anode copper. The project will have two distinct operating regimes:

- In Phase 1, the smelter will process a mix of Kamoa and Kakula concentrates, nominally in equal amounts. However, the mass percentage of Kamoa concentrate will vary between 20% and 70%.
- In Phase 2, the smelter will process only Kamoa concentrate, after the Kakula ore body has been exhausted.

The associated smelter acid plant will include a wet gas cleaning plant, drying and absorption system, conversion system, waste heat boiler, circulation water, acid tank farm, and acid loading facilities, wastewater treatment, etc. The plant will produce acid from high concentration SO_2 gas, linking the technology of partial dilution and pre-conversion with the process of double contact (four-conversion and two-absorption) acid production. In the conversion process, a waste heat boiler will be used to recover heat and produce saturated steam. The acid production process may be divided into the following three steps:

- Cooling and dust removal from the process gas from the dry ESPs.
- Conversion of SO_2 to SO_3.
- Absorption of SO_3 to form H_2SO_4.

The acid plant will need to be able to handle variation in the proportions of Kamoa and Kakula ore and concentrate. Design capacity will therefore be 783 ktpa of 100% H_2SO_4. The expected product grade is 98.5% H_2SO_4. Feed gas flowrate to the acid plant is expected to be in the range 71,000 to 85,000 NCMH at temperature of 300–320°C. Dust content will typically be about 215 g/NCM with excursions to a maximum of 1,000 g/NCM.

Table 17.5 Kamoa-Kakula Smelter Process Design Criteria – Major Items

Item	Units	Overall	
Processed Material Streams		Kamoa	Kakula
Blended Concentrate Operating Range	%	20% to 70% of total	80% to 30% of total
Concentrate Composition Range	% Cu	36.7	39.6 to 58.5
	% Fe	17.9	3.0 to 5.0
	% S	25.0	10.7 to 16.2
Major Sulfide Mineral Content	% (>5%)	Chalcopyrite 31%, Bornite 22%, Chalcocite 13%, Pyrite 8%	Chalcocite 61%, Bornite 6%
Concentrate Physical Characteristics		Bulk Density 2.21 t/m³, 12% H_2O, 80% <48 micron	
Direct Blister Furnace Blister Copper Quality	%	Cu>99.33, Fe<0.04, S<0.035	
Slag Cleaning Furnace Blister Copper Quality	%	Cu>99.00, Fe<0.34, S<0.25, Zn+Pb+Co+As+Ni<0.45	
Blister and Anode Copper Production Capacity	ktpa	500	
Anode Copper Quality	%	Cu>99.66, Fe 0, Ag+S+Pb+Ni+Se+As+Bi+O_2<0.33	
DBF Slag Major Constituents	%	Cu 16.8 to 21.3, magnetite 11.6 to 15.6	
SCF Slag Major Constituents and Bulk Density	%	Cu 2.8 to 3.6, magnetite 6.4 to 7.4, BD 2.65 tp	
Sulfuric Acid Product Quality	%	H_2SO_4>98.5	
Equipment Operating Conditions			
Smelter and Acid Plant Operating Time	Hours per Year	7400 (84.5%)	
Gaseous emission standards	mg/Nm³	SO_2 200, NO x 300, dust 5	
Concentrate Filter Cake Storage Capacity	Days	14	
DBF Operating Temperature	Degrees Celsius	Blister Copper 1,250, DBF Slag 1,300, Offgas<1,370	
SCF Operating Temperature	Degrees Celsius	Blister Copper 1,250, DBF Slag 1,350, Offgas 800 to 1,482	
SCF Operating Cycle	Hours per Cycle	4 hours Feed and Reduction, 2 Hours Settling and Discharge	
SCF Gas Phase Operating Regime	Ratio	CO to CO_2 Ratio is 1.02 During Settling, 5.38 During Reduction	
Tail Gas from Acid Plant and Desulfurisation	mg/Nm³	SO_2<170	

17.4.3 Kamoa-Kakula Smelter Design and Process Description

A high-level block flow diagram of the Kamoa-Kakula smelter is shown in Figure 17.3.

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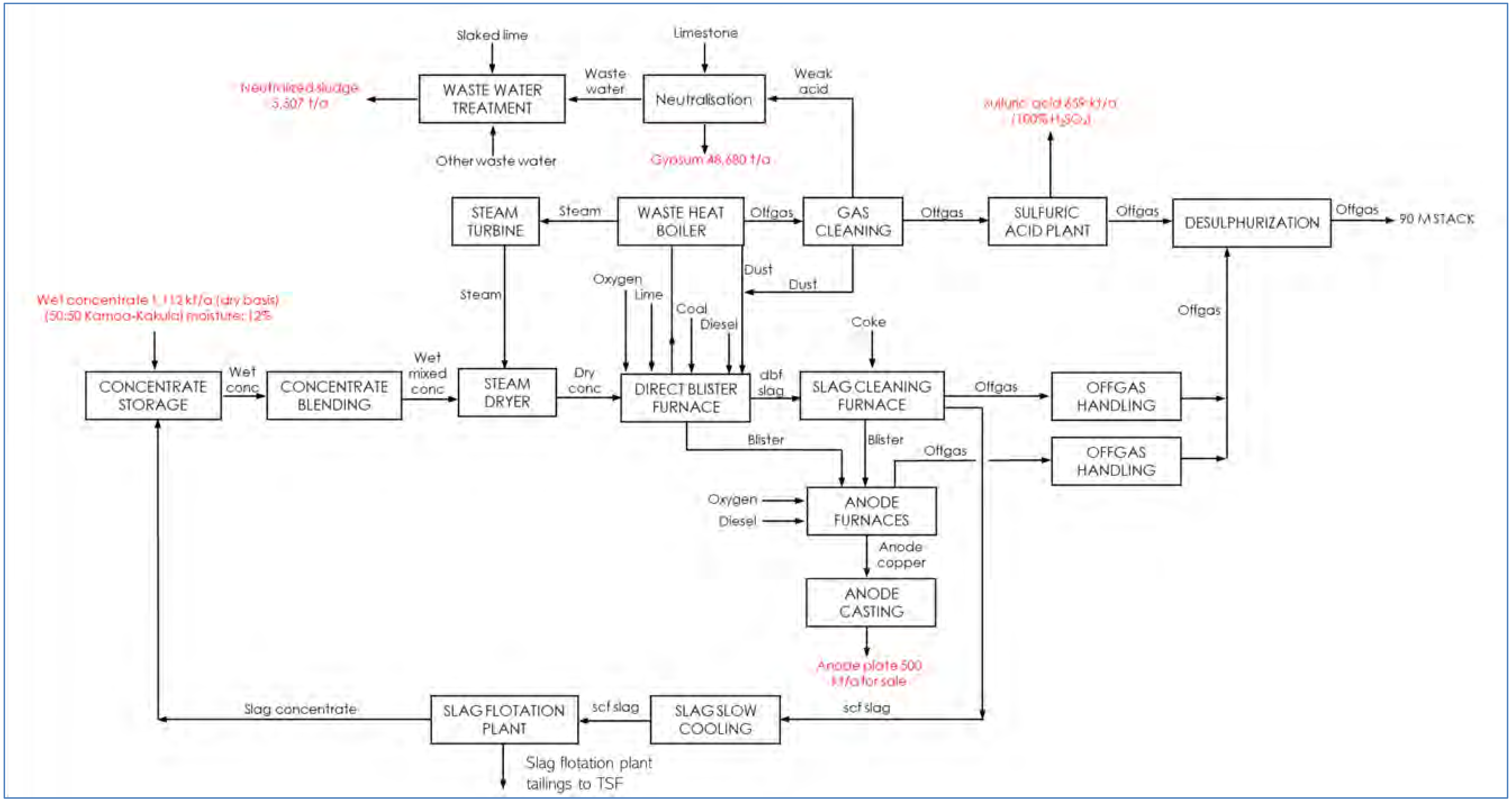
Figure 17.3 Kamoa-Kakula Smelter Block Flow Diagram



17.4.3.1 Trade-off studies

The following trade-off studies were completed before the selected flow sheet was finalised:

- Road vehicles were selected to transport almost all items in or out of the smelter. The only exception of significance is flotation tailings thickener underflow which will be pumped to the tailings storage facility.
- A wet concentrate filter cake storage facility with a rectangular configuration, and a gantry supported stacker reclaimer was selected in preference to multiple live circular stockpiles for concentrate, slag concentrate, anode furnace slag and coke.
- A circular blending stockpile on which the Kamoa, Kakula, and slag concentrates will be blended, with discharge conveyors beneath the stockpile was preferred to other alternatives studied.
- A waste heat boiler with flue dust collection was included in the DBF gas handling circuit.
- The volumetric flow of slag cleaning furnace offgas will be extremely variable in different parts of the batch operational cycle, making process control of the waste gas handling system very challenging. Dust recovery cyclones were selected ahead of electrostatic precipitators, notwithstanding their lower dust collection efficiency.
- Ground limestone was preferred as the active neutralising agent for the weak gas scrubbing process.
- A VSPA plant was selected instead of a cryogenic oxygen plant.
- Diesel generators compatible with the Kakula concentrator were specified for emergency power generation.

17.4.3.2 Smelter Raw Material Handling

Tailings thickener underflow will be pumped to a tailings storage facility and other materials will be transported in, or out, of the smelter area in road trucks. This includes concentrate filter cake which will be unloaded to two separate 200-tonne bins for Kamoa and Kakula concentrate. Concentrate will be discharged from only one bin at any time by belt feeders, transferring the filter cake to a single 800 tph conveyor belt equipped with a weightometer, and an automatic sampler to enable the quantity and chemical composition of the concentrates to be measured ahead of storage and blending.

The concentrates will be transferred from the conveyor belt to separate areas of the main 14 m high concentrate storage shed with dimensions of 165 m length by 42 m width and approximately 70,000 t or 14-days of storage capacity, using a tripper car. Flotation concentrate from the slag cleaning furnace slag recovery circuit will be transferred by truck, to a stockpile located in a dedicated storage shed.

Anode furnace slag will be transported to this area in trucks, before being broken up to a maximum top size of 100 mm using a mobile rock drill. Following breakage, the slag will be transferred to a bin, using front-end loaders. The material will be fed to a jaw crusher from which a belt conveyor will transfer material finer than about 30 mm to the AF slag, and SCF coke storage shed. Coke for Slag Cleaning Furnace reduction will also be stored in a separate area of this shed, to keep it dry.

Two gantry mounted scraper reclaimers will be used to transfer the main flotation concentrates onto a 600 tph transfer belt conveyor which will deliver the material to the blending area. Slag concentrate will be loaded into a hopper using FELs, then a 20 tph belt feeder with a weightometer will discharge this concentrate onto the same transfer conveyor as the two main concentrates.

The transfer conveyor will deliver the three concentrates to a 500 tph rotary stacker. There will only be two main ore flotation concentrates (Kakula and Kamoa) to be delivered sequentially, as well as a small quantity of slag flotation concentrate to be processed, so this simple blending arrangement is expected to be adequate. A reclaimer will withdraw concentrate from the stockpile and discharge it to a hopper and a 250 tph belt conveyor. Two emergency feed hoppers have also been specified over this conveyor, for use with front-end loaders if the blending stockpile stacker or reclaimer are offline.

Raw coal with variable moisture content will be delivered by truck to the raw coal stockpile. Capacity is about 550 tonnes, sufficient for seven days. FELs or trucks will be able to discharge coal into a hopper from which a vibrating feeder will transfer it to a high angle 15 tph belt conveyor equipped with a tramp iron magnet, and a weightometer. This conveyor will transfer coal to a reversible conveyor able either to discharge the raw coal into a 50 t bin ahead of a coal pulveriser, or directly into a coal crusher.

The crusher will be used to prepare coal to be fed to the hot gas generator via a 5 t feed hopper. A blower as well as some recycled coal mill exhaust gas will supply combustion and fluidising air to the hot gas generator from which the heated exhaust gas will be transferred to the 5 tph coal drying and pulverising mill. A weigh feeder beneath the raw coal bin will discharge directly into the coal mill. Dry pulverised coal and gas from the mill will be drawn through a bag filter by an exhaust fan. Pulverised coal with particle size less than 74 micron and moisture content of no more than 1.5% will drop out of the bag filter into a silo with about 70 t capacity of coal. In the silo the coal will be stored under nitrogen to prevent explosion or combustion, and the silo will be kept under slight negative pressure to limit escape of any coal dust into the adjacent working area.

Unslaked lime (quicklime) will be used to flux the slag in the DBF. It will be delivered in tankers to four tightly enclosed lime silos, each with about 1,130 m^3 live capacity, or sufficient for 3.5-days each. Each silo will operate under slight pressure and will be equipped with a bag filter, and an exhaust fan, to remove fine dust (that will be returned to the silos) and excess gas. Lime will be discharged from the silos by pneumatic conveyors using weigh flasks and operating under controlled conditions. Wall vibrators in the silos will be used to deal with any lime accretions inside the silos. Each pneumatic conveying system will be able to deliver 15 tph of lime to the DBF lime "target" bin that is also fitted with a bag filter, exhaust fan, wall vibrators and rotary valves to discharge lime to the furnace feed system.

A supplementary emergency lime addition system has also been specified in case the tanker truck delivery is delayed for any reason. Unslaked lime in bags will be delivered by truck to two 5 m^3 feed bins fitted with a bag filter and exhaust fan. Lime will be conveyed pneumatically to the target bin, using a dedicated system with weigh flasks.

17.4.3.3 Concentrate Steam Drying

Blended copper concentrate will be screened through a trash screen and then transferred via 250 tph belt conveyor and vibrating feeders to the concentrate steam dryer. The dryer will be capable of drying 200 tph of concentrate containing 12% moisture to 0.3%. During Phase 1 the worst case of feed to the smelter concentrate dryer is 199 tph of concentrate when the blend is 70% Kamoa, 30% Kakula concentrate.

The heat source for the dryer is the 2 MPa steam produced in the smelter and acid plant waste heat boilers. Dryer offgas will be routed to a bag filter to collect dust before being exhausted. to atmosphere. Dust collected from the bag filter will be conveyed by drag conveyor to the discharge bin of the steam dryer.

17.4.3.4 Direct Blister Furnace (DBF) Feed System

Dry copper concentrate, including the dust collected, will be transferred via rotary feeder to a 200 tph airlift system which will pass through an expansion chamber before transferring dry concentrate to the insulated and steam heated 300 m^3 DBF concentrate feed bin. Dust will be collected from the conveying air after the expansion chamber, using cyclones and a bag filter, and will be fed via rotary feeders to the concentrate dry charge bin. Clean conveying air will be exhausted to atmosphere.

Concentrate from the Dry Charge Bin will be fed to a 190 tph Loss in Weight feed system, consisting of screw feeders to a dosing bin.

From the pulverised coal silo, dry pulverised coal will be conveyed pneumatically under controlled conditions, at a rate of about 3 to 6 tph, to the 40 m^3 DBF coal feed bin located on top of the furnace. Multiple weigh flasks will be used to achieve the controlled coal transfer rate required, and both air and nitrogen will be used in this pneumatic conveying system to inhibit combustion or explosion of the pulverised coal. Most of the coal will drop out of suspension when it reaches the bin. The conveying gases will be routed through a bag filter before being recycled or exhausted to atmosphere. Coal recovered in the bag filter will be returned to the DBF coal feed bin via a rotary feeder.

Up to 15 tph of lime will be conveyed pneumatically to the 40 m^3 DBF lime feed bin, using a dedicated system with weigh flasks.

Lime (15 tph) and coal (2 tph) will be conveyed from the respective DBF feed bins using screw feeders and drag conveyors. Flue dust recovered from the DBF exhaust gas system will be conveyed pneumatically to a dedicated 140 m^3 Flue dust DBF feed bin. This dust will also be transferred using a 35 tph Loss in Weight feed system consisting of a dosing bin discharged by a screw feeder, to a drag conveyor. Lime, coal and dust will report with concentrate to a combiner duct before being despatched through a 240 tph air slide conveyor to the DBF concentrate burner.

17.4.3.5 Direct Blister Furnace

Concentrate treatment rate is expected to be in the range of 1 Mtpa to 1.2 Mtpa depending on the ratio of Kakula and Kamoa concentrates. The burner system needs to be able to accommodate the extremes of the possible range of Kamoa and Kakula concentrate ratios, prevalent in Phase 1. It has nominally been specified to process 240 tph of blended feed (including coal, lime and recycled dust as well as fresh concentrate) with 67,000 NCMH of oxygen enriched air being added to the burner to support combustion of the concentrate. The concentrate burner will also be equipped with a central diesel lance.

To maintain the heat balance in the furnace, pulverised coal is used as a supplementary fuel. The pulverised coal addition rate will be varied. More coal will be necessary when the percentage of Kakula concentrate is high, with most copper present as chalcocite (Cu_2S) instead of chalcopyrite ($CuFeS_2$).

In addition to the concentrate burner with pulverised coal addition, sixteen other burners will use 300 kg/h each of diesel fuel and a mixture of pure oxygen, oxygen-air or air depending on the zone location, to provide supplementary thermal energy. Six burners will be located in the reaction shaft, eight in the settler roof and two in the offgas uptake. These burners will be used for heating up the furnace as well as during production.

The DBF consists of a reaction shaft (internal dimensions 6.5 metre diameter by 7.8 metres high), settler (28 by 8.5 metre) and a 4.1 metre diameter uptake shaft.

Maximum depth of both the blister copper and DBF slag has been specified by the furnace designer to be only 1 metre. The slag needs to be accommodated in the DBF settler for about 8-hours before it can be transferred to the Slag Cleaning Furnace (SCF) for a batch reduction operation. The dimensions of the settler are set by these requirements and limitations.

The DBF water cooling arrangement includes the following:

- Three layers of water-cooling elements located at the top and in the body of the reaction shaft, to deal with the high heat load.
- Water-cooled copper or copper/steel cooling elements in the settler side wall, to protect the refractories, with the steel lining introduced in the lower part of the wall so that copper in the elements cannot come into contact with molten blister copper in the furnace.
- Cooling elements will also be installed on the roof of the settler near the rection shaft where there may be a considerable flow of hot gas. No water cooling has been specified in the section of settler roof close to the uptake.
- Where the settler transitions into the cylindrical uptake shaft there will again be high flowrates of hot gas. Zigzag water cooling elements are adopted. The uptake wall is a structure with masonry bricks in the middle of 11 layers of horizontal water jackets. The top of the uptake shaft will be made of refractory bricks and BIC water cooling elements, with an elastic structure for expansion absorption.
- Copper water cooling elements are used between the uptake and the boiler.

- The cooling water flow around the DBF body is 3,020 m^3/h. There are 25 water headers including six for the reaction shaft, 14 for the settler and five for the uptake shaft. All water headers have temperature and flow measurement and alarms included.

Since molten blister copper is permeable through the furnace bottom, to prevent copper leakage the DBF cooling system includes three furnace bottom air cooling fans (215,000 normal cubic metres per hour (NCMH), two fans operating and one on standby).

The DBF has nine blister copper tapholes. Launders heated with external burners will be used to transfer blister from three dedicated tapholes each, to the three anode furnaces. Average production rate of blister copper will be between 48 and 57 tph depending on the feed concentrate mix.

The maximum slag depth of 1 metre is more than for a conventional FSF, adding to the static pressure at the blister tapholes. Automatic tapping machines will be installed at the blister tapholes, in consequence.

The DBF has four slag tapholes. Slag containing about 18% Cu will be transferred every six-hours to the SCF via launders, for slag reduction or cleaning in a batch operation. Slag transfer will take two-hours, so three tapholes will be used for each tapping operation. An automatic tapping machine has been included in the design, although provision has also been retained to use a manual tapping system with oxygen lances and mud guns. Between 75 and 108 tph of slag will be produced, depending on the concentrate blend being processed.

17.4.3.6 Slag Cleaning Furnace (SCF)

The duty of the SCF is to recover copper from DBF slag that is expected to contain 17 to 21% Cu. The Kamoa SCF is designed to produce slag containing 2.8 to 3.6% Cu. This will later be sent to the slag flotation circuit from which the flotation tailings will ideally contain only 0.2 to 0.4%. Very strong and effective reduction is required, in this first step of producing a discardable waste product. The SCF is therefore a critical equipment item for the project.

The main technical features of the SCF are summarised below.

The SCF will operate in batch mode. The operational cycle specified involves two-hours slag transfer with some reduction occurring, 2-hours reduction, two-hours settling and two-hours of slag tapping. The necessity to design the SCF to handle DBF slag produced from a wide variation of copper concentrate composition as well as the very different operational parameters of the reduction and settling phases of the SCF batch cycle made detailed design of the SCF and its offgas handling system very challenging. The waste gas flowrate is shown as varying between 32,000 and 82,000 NCMH, for example.

The dimensions specified for the rectangular furnace are 18 metres length by 10 metres width by 3.5 m height. The furnace is equipped with two independent sets of three electrodes each, with two 15 MVA transformers. This arrangement will allow different power input and reduction strategies to be applied in the front and rear sections of the furnace hearth.

SCF refractories will be compressed using a steel structure with springs and stops at the top of each column. A horizontal roof using hung bricks and equipped with temperature and pressure monitoring as well as inspection holes has been specified, with several feed inlets for example for coke, flux and top blown reduction lances. Sealing devices have been included around the electrodes at roof level. The hangers of the roof bricks will be made of heat-resistant stainless steel.

The bottom of the furnace will be air cooled to ensure no leakage of permeable molten blister copper through the three layers of refractory brick. Horizontal copper plate coolers have been included in the furnace freeboard sidewall above the bath. Elsewhere, side walls will be water cooled in critical sectors where the inner wall is in contact with molten material. Taphole areas will also be water cooled using a system with a higher rate of heat transfer than the sidewall cooling. In addition, corrosion and erosion resistant bricks have been specified at the slag line since the slag is far more corrosive and erosive than blister copper.

The electrodes will consist of cylindrical steel casings that will be filled with baked paste. Blocks of fresh paste will be added daily through the top of the electrodes and will melt then bake as the paste and casing at the bottom of each electrode are consumed. It will also be necessary to periodically weld new sections of electrode casing onto the top of each electrode.

Significant quantities of coke will be added using seven 4.5 m^3 bins and seven 4 tph feeders, with an eighth bin and feeder being used to add crushed anode furnace slag from the smelter feed preparation area to the SCF. In principle, small quantities of lime flux could also be added to the SCF if required. The coke will act as reductant, and this will in turn generate carbon monoxide gas (CO). This needs to be removed before desulfurisation of the SCF offgas. A post combustion chamber (incinerator) as well as combustion and dilution blowers have been specified to oxidise all CO before the SCF offgas is sent to the evaporative cooler and then to the weak gas scrubber dedicated to the slag cleaning furnace.

The slag cleaning furnace has been sized to handle the maximum amount of DBF slag expected when the ratio of Kamoa to Kakula concentrate is 70:30. The SCF needs to be able to process 2,582 tonnes per day of incoming slag from the DBF. The DBF slag composition will vary considerably depending on the Kamoa to Kakula ratio. The SCF electrical and mechanical design needs to cater for this variation, which will influence the slag resistivity.

400 mm minimum slag layer has been specified to prevent short circuiting, with about 1 metre of additional variable slag depth. This will enable electrode power to be kept within a reasonable operating range. Consequently, the bath in the SCF needs to have an area of about 180 square metres.

Six slag tapholes have been included in the end wall of the furnace, at two different levels, 750 mm and 1,050 mm above the blister copper taphole. These tapholes will transfer molten slag to ladles, via launders. Maximum slag level in the SCF is 2,524 mm above the blister taphole. Provision has been made for automated and manual SCF slag tapping, and for tapping from the multiple tapholes at different heights and locations, to remove the cleaned slag as quickly as possible before commencing a new batch cycle.

SCF slag will be moved to a holding area using ladle transporters. The ladles will be allowed to cool naturally for 8-hours to allow blister copper and copper oxide mineral grains to coalesce, making them more amenable to recovery by flotation. After this it is expected to be safe to spray water onto the ladles. After water cooling for 54 additional hours, the ladles will be emptied out and returned to the SCF. Over 200 ladles have been specified for this duty. The subsequent processing steps are described in the slag flotation circuit section of this report.

Two tapholes have been included for blister copper, which will be transferred to a dedicated anode furnace. The blister copper level will cycle between 50 mm and 300 mm above the tapholes, due to the batch cycle operation of the SCF. In an emergency, blister copper from the SCF can also be discharged into one of the other anode furnaces or into slag ladles.

A heat balance around the SCF indicates that input power will be 22 MW. Each three-electrode cluster will need to deliver 11 MW and a 30 MW power supply has been specified. Maximum electrode current has been calculated to be 49 kA, with maximum current density of the 1,200 mm diameter electrodes being 4.4 A/cm^2. Most electrode pastes can operate at current densities up to 5 A/cm^2.

17.4.3.7 Anode Furnace (AF) Refining and Casting

There will be three rotary anode furnaces (AFs) each with 660 tonne nominal capacity. Molten blister copper will be added via the launders from the DBF or ladle from the SCF to an AF, then compressed air and diesel will be added to promote oxidation while also maintaining the necessary operating temperature. Solid reductant coke will also be used in the AFs when required, to be fed from the unloading and storage area using a nitrogen based pneumatic conveying system. SCF blister copper ladles transported by hot metal crane will be used to ensure that the AFs remain in production even if there is no DBF blister copper available at short notice.

Anode furnace slag will be cooled, crushed and recycled to the SCF as described earlier. AF off gas will be cooled in the hood to 400 to 600°C, and then will pass through an evaporative cooler, before reporting to the weak gas scrubber. Anode copper is expected to be very high-grade (>99.7% Cu), consequently anode furnace slag will probably contain significant concentrations of copper, as well as impurity elements.

Anode casting will be done using two 110 tph twin wheel 18 mould casting machines. Barium sulphate release agent will be used. Steam hoods above the casting machines will be used to capture vapour that will be exhausted directly to atmosphere. Dedicated “take-off systems” will be used to remove anodes from the casting wheels, after they have solidified.

A shaft furnace with capacity of 30 tph of production and 180 tonnes of holding capacity will use diesel oil as fuel and will be utilised to melt down anode and cathode scrap. The product will be anode copper, to be refined. Shaft furnace offgas will be air cooled to 150°C before being sent to the fugitive gas bag filter and desulfurisation system.

17.4.3.8 Cooling Water Systems

The furnace cooling water system is mainly to supply cooling water for DBF, SCF, anode furnace, fugitive gas fans, dust collecting fans, and other equipment, and provide emergency water for the emergency water system (capacity: 6,218 m^3 /h).

17.4.3.9 DBF Waste Gas System

DBF waste gas contains a significant amount of waste heat that is recoverable in a waste heat boiler, after which it will be necessary to recover fine concentrate dust from the gas stream before sending it to desulfurisation.

Air dilution into the WHB amounts to about 25% of the DBF waste gas flowrate, or 11,000 NCMH. The WHB will therefore receive about 59,000 NCMH of gas at a temperature of 1,370°C and in the WHB the gas temperature will be reduced to about 350°C. Dew point for the specified gas composition range was calculated to be about 240°C, and temperatures therefore always need to exceed this. The DBF offgas will contain significant amounts of dust and the WHB has therefore been specified with abrasion resistant linings in the inlet gas section of the unit.

65 tph of steam from the WHB at a pressure of 6 MPa will be sent to #1 steam turbine and used for waste heat power generation.

Approximately 59,000 NCMH of gas exiting the WHB at 350°C will be sent to a dust-settling chamber, with dust that drops out of suspension here being transferred via drag conveyor and rotary feeder. This dust will include some lumps and will therefore be crushed in two dedicated 10 tph crushers before being routed to the DBF dust conveying system.

After the dust-settling chamber, the partially de-dusted gas will report to two electrostatic precipitators (ESPs) equipped with air heaters and vibrators to free accretions in the discharge chutes. Dust collected in the ESPs will be removed by drag conveyor before being transferred via rotary valve to another drag conveyor that is used to transfer the combined dust to a bin from which it will be pneumatically conveyed to the 140 m^3 DBF Flue dust feed bin.

About 73,000 NCMH of cleaned DBF gas at 300°C from both ESPs will be transported using two exhaust fans (one operating, one standby) to the primary gas scrubber at the inlet of the acid plant.

Provision has been made to route DBF gas to the fugitive gas system and then to atmosphere after removing some dust in the settling chamber, if either the WHB or the ESPs fail.

17.4.3.10 SCF Waste Gas Handling System

During the SCF charging and reduction periods, about 127,000 NCMH (design) of diluted SCF gas from the incinerator (used to oxidise CO) is routed to an evaporative cooler in which process water is injected to cool the gas to 200°C before it reports to a cyclone to remove any residual dust.

During the SCF settling period, dilution air will be introduced to increase the flow to about 61,000 NCMH and gas leaving the incinerator is only at a temperature of 150°C, so no evaporative cooling is needed.

Dust from both the evaporative cooler, and the cyclone use drag conveyors and rotary valves to transfer the dust to portable bins that are transported to the DBF system. The quantity is less than 2 tph. Between 70,000 and 200,000 NCMH of clean gas from the cyclone will report to centrifugal exhaust fans and then to the weak gas desulfurisation scrubber.

17.4.3.11 Anode Furnace Waste Gas System

About 17,000 NCMH of dusty gas from the three Anode Furnaces (AFs) at temperatures between 400 and 650°C reports to evaporative coolers in which the gas temperature is reduced to 200°C.

Small quantities of dust from the AF evaporative coolers are collected via rotary feeders into dust bins to be transported to the DBF dust collection and recycling system. The gas from the evaporative coolers is combined and exhaust fans are used to transfer this gas stream (now about 72,500 NCMH) to the weak gas scrubbing system.

17.4.3.12 Fugitive Gas Handling

Fugitive gases will be collected at more than 20 blister and slag tapholes at the DBF and SCF, as well as the Shaft Furnace (SF). In addition, DBF slag inlets to the SCF, charge ports, blister ladles, launders and slag ladles will also be equipped with fugitive gas hoods connecting to ducts and the extraction system.

All gas exhaust ducts will be equipped with motorised butterfly valves to isolate them when extraction is not required. The maximum gas flow of the combined fugitive gas system is 188,698 NCMH in the temperature range 80 to 90°C. The combined flow will be routed through a bag filter and then will be exhausted to weak gas scrubbing. The exhaust fan is equipped with a variable speed drive given the considerable variation in flowrate. Dust collected will be recovered and stored in hoppers. These will periodically be transported to the DBF for recycle to smelting via the DBF dust conveying system.

17.4.3.13 Weak Gas Desulfurisation and Effluent Treatment

There are five gas streams to be treated in the gas desulfurisation system, which are AF process gas, SCF process gas, fugitive gas, Acid Plant tail gas and coal preparation gas. Tables in the appropriate volume of the Nerin report summarise the estimated gas flowrates, temperatures and compositions clearly. A few key points are summarised below:

- AF and SCF process gas contain dust and are scrubbed and de-dusted separately before being combined with the fugitive gases.
- The most common weak gas desulfurisation process has been selected, using "Milk of lime" addition and oxidation to generate a solid gypsum product.
- This process permits wide fluctuations in gas volumes and concentrations and the reagents can easily be obtained.

- Gypsum is already being produced elsewhere in the project in the wastewater treatment circuit, so some additional production is not going to be problematic.
- After preliminary dust removal, the gases will be routed to a mixing chamber then blowers transfer the total gas stream to the scrubbing tower. Most of the slurry with which the gases are contacted is recirculated through the tower, but a bleed stream will be sent to the gypsum recovery circuit.
- The de-dusted gas entering the desulfurisation tower will be contacted with limestone slurry. After absorption, about 92% of sulfur dioxide is expected to be removed from the gases in gypsum.
- Following removal of liquid droplets in a wet electrostatic separator, the scrubber tail gas will be exhausted to atmosphere through a dedicated stack.
- Gypsum in the slurry stream will gradually be concentrated. After filtration, a vacuum belt filter will be used to produce gypsum filter cake containing about 15% moisture.
- Limestone slurry needed for the process will be prepared in an upstream system.

17.4.3.14 Sulfuric Acid Plant (SAP) Wet Gas Cleaning

This will comprise a primary reverse jet scrubber, gas cooling tower, secondary reverse jet scrubber, and two-stage wet electrostatic precipitator. This is a well proven and reliable arrangement that can tolerate turn down in gas flowrate from 100% to 50%.

Dust, metallic fumes, acid mist and water vapour will be removed from the gas in the scrubbing system. The gas is quenched by adiabatic saturation. Sodium silicate will be added to the primary scrubber to remove fluorides.

Further gas cooling and water condensing will be achieved in the gas cooling packed tower, where enough water will be condensed from the SO_2 gas stream to allow production of high-strength H_2SO_4 later in the process.

The second gas scrubber after gas cooling is used primarily to polish the gas by removing any remaining particulates. After this, the clean gas will be routed through electrostatic mist precipitators to remove any remaining particles of acid mist, metallic fumes or solids.

Heat is removed from the system by cooling the recirculated weak acid using plate coolers. The expected gas outlet temperature is 40°C. Weak acid solution leaving the scrubber will contain about 20% acid. The multiple stages of impurity removal are essential since any remaining impurities in the gas will most likely report to the final product acid.

A bleed stream of dilute acid, from the primary scrubber outlet, will be processed by settling and filtration of the settler underflow to recover a filter cake containing lead and remove it from the system. Filtrate as well as nearly all of the settler overflow will be recycled to the primary scrubber.

The wet gas scrubbing section is mainly constructed of FRP hence the cooling and water/weak acid circulation systems are extremely important to keep the operating temperature down. Necessary backup systems and equipment have been incorporated.

17.4.3.15 Sulfuric Acid Process

The concentration limit in feed gas to a conventional double contact sulfuric acid plant is 12% SO_2. This will always be exceeded in the Kamoa-Kakula acid plant, even with the smelter operating on concentrate produced from the lowest possible Phase 1 percentage of Kamoa ore (20%). SO_2 content of the gas will be between 21.75% and 32.85%.

Dilution of feed gas followed by proven pre-conversion technology ahead of a conventional double-contact process has been proposed. Feed gas flowrate to the acid plant is therefore expected to be in the range 71,000 to 85,000 NCMH at temperature of 300–320°C. Dust content will typically be about 215 g/NCM with excursions to a maximum of 1,000 g/NCM. Design production capacity will be 783 ktpa of 100% H_2SO_4. The expected product grade is 98.5% H_2SO_4 and overall SO_2 recovery is expected to be 99.9%.

The following steps are included in the Sulfuric Acid plant:

- Cleaned feed gas is sent to one stage of drying, counter-current with cooled 96% sulfuric acid and dried air in a packed tower followed by two stages of absorption and cooling. The acid plant main gas blower is located between drying and absorption.
- Conversion of SO_2 to SO_3 occurs in a four-pass converter (packed bed tower) in the presence of oxygen and catalyst, during which considerable heat is generated. Without a pre-conversion process, the exit temperature of the first stage of conversion would exceed the allowable temperature limits for the catalyst and converter vessel.
- In the pre-converter, a portion of the SO_2-rich gas from the drying tower is mixed with atmospheric dilution air, heated, and sent to a pre-converter containing catalyst. The pre-converted gas stream is then re-combined with the remaining dried feed gas resulting in gas to the first stage of conversion containing about 16% SO_2, as well as some SO_3 (which acts as a heat sink going forward) from the pre-converter.
- Conversion of remaining SO_2 to SO_3 takes place in a four-pass converter in the presence of oxygen and vanadium catalyst. This conversion reaction consumes oxygen and produces heat. The extent of the conversion reaction in the converter passes is limited by equilibrium as the gas temperature rises due to reaction heat and the formation of SO_3. The extent or efficiency of conversion is increased by carrying the reaction out in successive catalyst passes with partial cooling between the passes. The overall conversion is further enhanced by double absorption.
- Conversion will only commence if process gas is heated to the auto ignition temperature before contacting the first pass (stage) catalyst. This is done using a diesel pre-heater and then by recovering heat generated by the exothermic conversion reaction. Reaction heat is recovered by the generation of high-pressure steam in a waste heat boiler from which process gas exiting the boiler is recycled to the first stage of the converter. Heat exchangers and additional waste heat boilers are used to control the temperature in the remaining stages of the converter.
- Cooled process gas leaving the conversion process contains high concentrations of SO_3 which is absorbed into a stream of 98.5% (w/w) H_2SO_4 in the two stages of absorption.
- Process gas leaving the absorption passes through mist eliminator elements before being discharged to the desulfurisation system.

- Medium pressure saturated steam (7 to 11 tph at 2.5 MPa pressure) from the first waste heat boiler and low-pressure steam (6 to 12 tph at 0.6 MPa) will be sent to the waste heat power generation system of the smelter.
- The proposed Acid Plant configuration, 3 +1 double absorption is illustrated in Figure 17.4 below, with the key being as follows:
  - Blue Circle, three initial contacts
  - Green Circle, one additional contact
  - The Blue + Green circles comprise the double absorption.

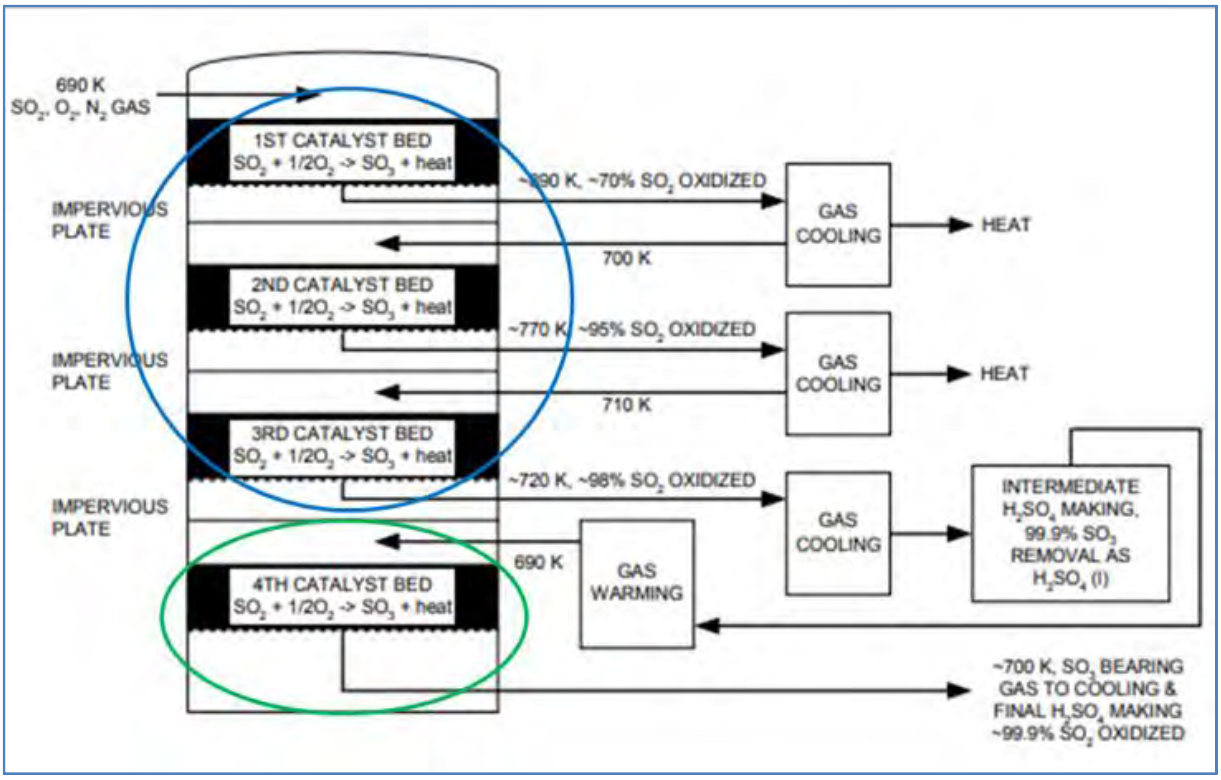
17.4.3.16 Acid Plant Circulating Water

Acid Plant circulation water is mainly fed to the gas cleaning, drying and absorption, and conversion systems as the equipment cooling water. The water supply capacity is 7,447 m³/h and pressure is about 0.32 MPa.

17.4.3.17 Acid Storage and Despatch

Eight by 10,000 tonne tanks have been included in the design to provide 30-days storage of 98.5% sulfuric acid. From these storage tanks, acid will be pumped to tankers for sale.

Figure 17.4 Kamoa-Kakula Acid Plant Proposed Configuration



17.4.4 Kamoa-Kakula Smelter Services Requirements

17.4.4.1 Oxygen Supply and Distribution System

The oxygen plant supplies oxygen for the DBF furnace, anode furnaces and electric slag cleaning furnace. The Vacuum Pressure Swing Adsorption (VPSA) process specified has the advantages of simple procedures, mature technology, fast start-up if the plant needs to be shut down temporarily, wide controllable range, easy adjustment, and low specific electricity consumption. The specified oxygen production rate is 38,000 NCMH at 93% purity.

The first step in the VPSA process is adsorption. Incoming air is filtered to remove particulates and then is pressurised to 35-45 kPa with a blower before being passed through an adsorbent bed. Water, carbon dioxide and nitrogen are adsorbed while nearly all of the oxygen exits the vessel as enriched gas containing 93% O_2. After an oxygen buffer tank, two booster compressors are used to increase the pressure of the oxygen stream, to 1.2 MPa and 0.25 MPa respectively.

After adsorption is complete, the pressurised oxygen is stored in two pressure vessels from which it is routed to the point of utilisation in the smelter, or higher-pressure oxygen can be mixed with the lower-pressure gas stream. A vacuum is then applied to the loaded adsorbent bed in a counter-current direction, causing the impurities to be released and completely regenerating the adsorbent. Pressurised oxygen is then injected into this vessel to increase the adsorption tower pressure to operating pressure. The two adsorption towers have now completed a full adsorption-regeneration cycle and are ready for the next cycle to commence.

The oxygen plant requires a circulation water system supplying 1,200 m^3/h at 0.4 MPa pressure.

17.4.4.2 Nitrogen

The nitrogen plant supplies gas to several units in the smelter in which it would be inadvisable to use air containing combustible oxygen for conveying, cooling, dust extraction etc. This includes certain areas of the DBF furnace, anode furnace, steam dryer, pneumatic conveying and pulverised coal preparation.

Pressure swing adsorption (PSA) has been specified for this circuit. Nitrogen is produced by adsorbing oxygen, carbon dioxide and water vapour onto an adsorbent bed under pressure. Approximately 11,000 NCMH of nitrogen will be produced at 99% purity and 0.6 MPa pressure.

For most of the applications, nitrogen pressure needs to be reduced to atmospheric pressure. This will be done after the nitrogen storage/buffer tank and in addition nitrogen compressors after the buffer tank will produce two grades of nitrogen at 2.5 MPa and 1.2 MPa, for different applications. The impurities are then desorbed from the adsorbent bed at atmospheric pressure, releasing them to atmosphere and regenerating the adsorbent in the bed.

Two towers are operated, one for adsorption while the other is in regeneration mode. Air is fed continuously to the plant.

17.4.4.3 Waste Heat Power Generation

Two steam turbines (one back-pressure turbine and one condensing turbine) have been specified, with capacity equivalent to the amount of steam produced by the waste heat boilers.

The back pressure turbine is used to convert the thermal energy of high-pressure steam, from the DBF waste heat boiler, into electrical energy, while also discharging medium-pressure saturated steam.

Steam discharged from the back pressure turbine is combined with medium-pressure steam from the sulfuric acid first conversion stage waste heat boiler. This steam is used in the concentrate steam dryer, with excess medium-pressure steam being condensed to enable its energy to be converted to electricity. The steam is physically converted into condensate.

Low-pressure steam from the WHB in the second converter stage of the acid plant is used plant wide by low-pressure steam users supplemented, if necessary, by discharge steam from the back pressure turbine.

17.4.4.4 Boiler Water Supply

The boilers require a steady supply of 39.5 tph of de-mineralised water. The de-mineralised water production facilities have been specified conservatively as two by 30 tph units. This water is produced in a two-stage reverse osmosis plant, to remove salts from the incoming process water.

The reverse osmosis process flow consists of the following steps: Clarified water → clarified water tank → clarified water pump → ultra filtration → first stage guard filter → first stage high-pressure pump → first stage reverse osmosis → intermediate water tank → intermediate water pump → second stage high-pressure pump → second stage reverse osmosis → electro de-ionisation (EDI) booster pump → EDI guard filter → EDI water treatment system → de-salinated water tank → desalinated water pump → deaerator.

In addition, the waste heat boilers need a supply of de-oxygenated water. This is produced by de-aeration of condensate from all steam users. The de-oxygenated water is then pumped under pressure to the waste heat boilers and other users.

17.4.4.5 Compressed Air System

This system is used to produce compressed air for the DBF furnace, anode furnace, steam dryer, pneumatic conveying, pneumatic instruments and valves, pulverised coal preparation and other process devices.

Approximately 927 NCM per minute can be supplied, at 0.5 to 0.8 MPa pressure. All compressed air will be dried, to one of two different standards, according to the requirements of the various users. Most of the compressed air is generated using centrifugal compressors, with a screw compressor for instrument air and as a backup for the main compressors. Drying is mainly achieved by freezing, with instrument air being dried by micro-heat regeneration.

17.4.4.6 Diesel Storage and Reticulation

Two 1,000 m^3 tanks have been specified for diesel storage, which is sufficient for about 10-days of consumption in the smelter. The diesel oil storage and refuelling station consists of the tank area, unloading area, refuelling area, pump shed, fire-fighting equipment room, distribution room for firefighting and ablutions.

A dedicated third 1,000 m^3 tank has been specified for diesel storage and supply to the DBF concentrate burner.

17.4.4.7 Water Treatment Facilities

Incoming water contains 26 to 42 mg/l of suspended solids. This needs to be reduced for many process duties in the smelter. Process water consumption in the smelter is expected to be approximately 14,331 m^3/day, hence the water treatment facilities have been specified to process 19,200 m^3/day.

Water treatment consists of purification by means of mixed flocculation, sedimentation and filtration. Not all duties in the smelter require all these steps to be applied.

Raw water is sent to a static pipeline mixer and aluminium sulfate coagulator is added at a controlled rate. Water is discharged to a settling tank in which coagulation occurs, aided by particles present. Most flocs precipitate in the settler, with small flocs being removed from the settler overflow by filtration. The filtrate contains only 3 mg/l of suspended solids.

Filtrate flows by gravity to the process water and fire water ponds. The total volume of these ponds is 5,800 cubic metres, sufficient for 8-hours consumption. Three process water supply pumps (two operational, one standby), can deliver a maximum of 1,300 m^3/h at 0.3 MPa, although the average flowrate is much less than this.

Potable water is delivered to the plant by the Kamoa operations team, so there is no potable water treatment plant within the smelter scope of supply.

There is also a recycled water circuit capable of receiving and distributing 2,478 m^3/day of water, mainly used in the desulfurisation circuits for the smelter, and to provide make-up water for the slag flotation plant.

Sewerage from the smelter ablution blocks is pre-treated in a septic tank before being routed to a small domestic sewerage treatment plant consisting of conditioning tank, pumping, clarification zone, anoxic zone I, aerobic zone I, anoxic zone II, aerobic zone II, settling zone, filtration zone and disinfection zone.

Stormwater is contained in the smelter according to the potential contamination (e.g., by acid) possible in each area. Potentially contaminated water is routed to an effluent treatment station whereas uncontaminated stormwater is collected in stormwater drains that discharge into lined ponds, and a dam with overall capacity of 80,000 cubic metres. The effluent removal circuit for stormwater includes aeration to precipitate arsenic, caustic soda addition, flocculation, settling and filtration.

Table 17.6 Projected Smelter Consumables

China Nerin Engineering Co., Ltd.			KAMOA COPPER SA		Document ID.:	1013-NRN-4000-PR-TEN-0002
			KAMOA COPPER SMELTER PROJECT		Rev No.	B
			Consumption of Raw Materials, Fuels and Main Auxiliary Materials		Date:	20/05/2022
S.N	Designation	Unit	Total (/a)			Note
			70% Kamoa	50% Kamoa	20% Kamoa	
1	Raw Material					
	Copper Concentrate	t	1,203,506	1,111,613	998,053	Dry basis
2	Product					
	Anode	t	583,915	583,314	582,506	
	100% H_2SO_4	t	782,642	658,940	504,210	
3.1	Fuel and Power					
	Light diesel	t	21,489	29,273	33,416	
	Fine Coal	t	–	3,524	13,973	Dry basis
	Coke	t	24,997	19,240	12,000	Dry basis
	Coke powder bases reductant	t	6,930			Dry basis
	Process water	10^3m^3	4,424.86			
	Demineralised water	10^3m^3	45.5			
	Electric Power	k-kWh	716,000			
3.2	Auxiliary materials					
	Limestone powder		31,446			Including 85% $CaCO_3$, used in Limestone Emulsifying and Desulfurisation
	Lime powder	t	1,177			Including 70% $Ca(OH)_2$, Used in Limestone Emulsifying
	Refractories	t	138			
	Refractories bricks	t	1,446			
	Electrode paste	set	500			
	Thermocouple	t	20,038			
	Barium sulfate	t/d	1,232			
	Sodium hydroxide	t/d	380			
	(DT CR)	t/d	0.12			Used in Storm Water Treatment Plant
	Polyacrylamide	t	0.008			Used in Storm Water Treatment Plant
	Polyacrylamide	t/d	3.47			Used in Effluent Treatment Plant
	(PFS)	t	0.12			Used in Storm Water Treatment Plant
	Quick lime	m^3	56,240	66,600	74,000	Dry basis
	Catalysis	t	75	63	48	Used in SAP Gas Conversion
	Grinding steel ball	t	500.9	444.9	384.5	
	Impeller, cover plate	t	125.2	111.2	96.1	
	Liner	t	219.2	194.6	168.2	
	Belt	t	0.9	0.8	0.7	
	Pump wearing parts	t	6.3	5.6	4.8	

China Nerin Engineering Co., Ltd.			KAMOA COPPER SA		Document ID.:	1013-NRN-4000-PR-TEN-0002
			KAMOA COPPER SMELTER PROJECT		Rev No.	B
			Consumption of Raw Materials, Fuels and Main Auxiliary Materials		Date:	20/05/2022
	Filter cloth	t	1.9	1.7	1.4	
	Z-200	t	62.6	55.6	48.1	
	MIBC	t	81.4	72.3	62.5	
	Na_2S	t	156.5	139.0	120.2	
	$NaHCO_3$	t	375.7	333.6	288.4	
	Engine Oil	t	31.3	27.8	24.0	
	Grease	t	62.6	55.6	48.1	
	Sodium Silicate (100%)	t	146.8	208.2	258.7	Used in Wet Gas Cleaning
	Ironvitriol	t	405.26			Used in Effluent Treatment Plant
	Filter bag	t	10,617			
	Oxygen gun (Buring copper tapping hole)	set	27,747			SS304, DN20, Φ3mm, L=6m
	Plugging gun	set	5,179			
	Mud	m^3	63			
	Cover filter tube	m^2	1,550			
	Slow-release scale inhibitor	t	145.97			Used in Circulation Water System
	Biocide-algaecide	t	14.76			Used in Circulation Water System

17.5 Kamoa-Kakula Slag Concentrator Circuit

17.5.1 Introduction

This section of the plant is designed for Phase 2, to process 2,078 tonnes per day (about 87 tph) of FSF slag containing 1.62% Cu. The corresponding Phase 1 tonnages and grades are:

- 1,506 tpd (63 tph) of SCF slag containing 2.8% Cu for a Kamoa: Kakula mass split of 20% to 80%.
- 2,033 tpd (85 tph) of SCF slag containing 3.55% Cu for a 70% to 30% split of Kamoa to Kakula.

Consequently, the slag flotation plant will be utilised less in Phase 1, to match the lower average feed rates applicable in all circumstances.

17.5.2 Kamoa-Kakula Slag Concentrator Basis of Design

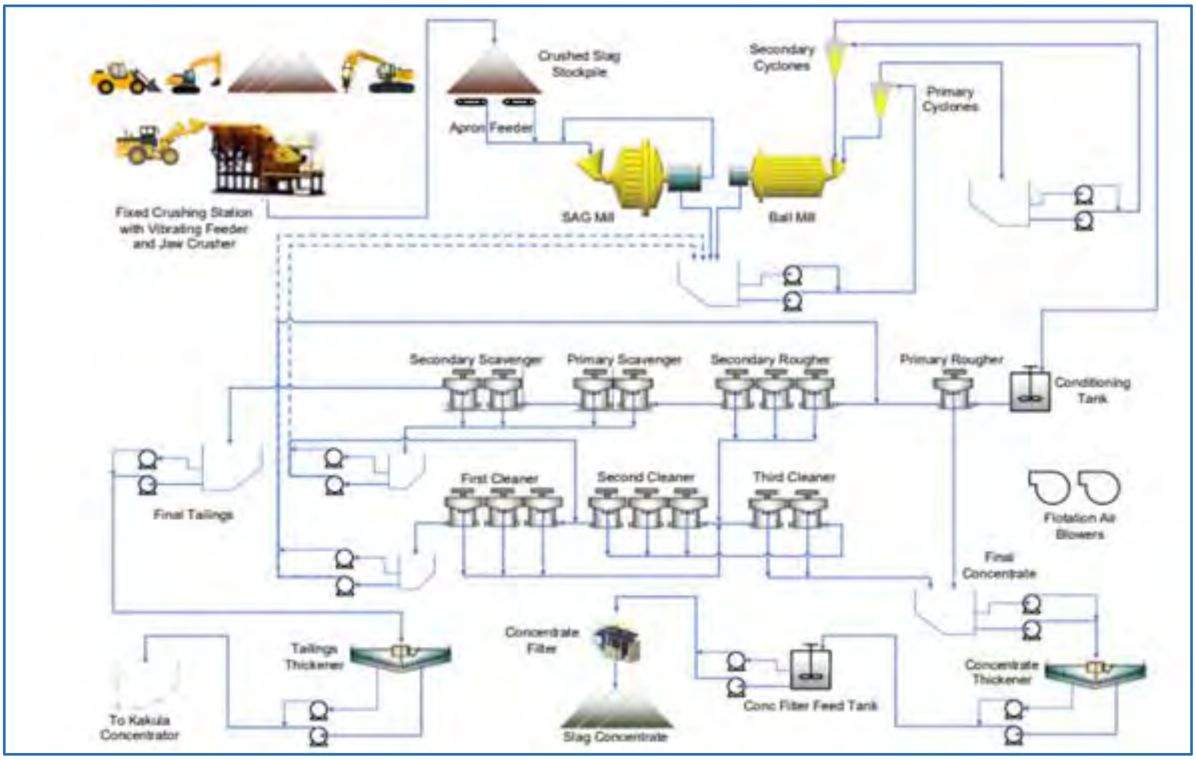
Table 17.7 Kamoa-Kakula Slag Concentrator Major Design Criteria

FLOTATION		
Flotation configuration	2 stages of rougher-2 stages of Scavenger-3 stages of Cleaner	
Minimum Step Height	0.6	m
Primary Rougher		
Feed rate	87.5	t/h
Pulp Volume	135.6	m^3/h
Pulp Density	44.2	% w/w
Feed Particle Size	45	µm
pH	8-10	
Residence Time, Selected	13.7	min
Feed Rate Scale-up Factor	1.1	#
Aeration per Cell, Maximum	1.2	$m^3/(m^2\ min)$
Flotation Air Pressure	45-53	kPa(g)
Primary Rougher Concentrate	4.21% mass yield @ 27% Cu to final concentrate	
Secondary Rougher		
Feed rate	107.8	tph
Pulp Volume	227.3	m^3/hr
Pulp Density	35.4	% w/w
Residence Time, Selected	24.5	min
Secondary Rougher Concentrate	26.4% yield @ 8% Cu, 65% Cu recovery	%
Destination	Second Cleaner Flotation Feed	
Primary Scavenger		
Feed rate	85	tph
Pulp Volume	177.8	m^3/h
Pulp Density	35.6	% w/w
Residence Time, Selected	20.9	min
Primary Scavenger Concentrate		
Yield	5.42 @ 6% Cu, 10% Cu recovery	%
Destination	Scavenger concentrate hopper	
Secondary Scavenger	80	t/h
Pulp Volume	166.4	m^3/h
Pulp Density	25.8	% w/w
Residence Time, Selected	28.3	min
Secondary Scavenger Concentrate	4.1% mass yield @ 4% Cu, 5% recovery	
Destination	Scavenger concentrate hopper	
Final Tailings	87.35% of feed mass at 0.5% Cu, 13.4% of Cu lost	
First Cleaner		
Feed rate	40	t/h
Pulp Volume	124.6	m^3/h
Pulp Density	26.1	% w/w
Residence Time, Selected	16.4	min
First Cleaner Concentrate	17.6% mass yield at 12% Cu grade, 65% Cu recovery	
Destination	Second Cleaner Flotation Feed	
Second Cleaner		
Feed rate	41.7	t/h
Pulp Volume	116.8	m^3/h
Pulp Density	28.4	% w/w
Residence Time, Selected	17.5	min
Second Cleaner Concentrate	11.5% mass yield @ 17% Cu grade, 60% Cu recovery	
Destination	Third Cleaner Flotation Feed	
Third Cleaner		
Feed rate	10.0	t/h
Pulp Volume	26.3	m^3/h
Pulp Density	30	% w/w
Residence Time, Selected	51.7	min
Third Cleaner Concentrate	7.8% mass yield @ 19.3% Cu grade, 46.6% Cu recovery	
Destination	Final Concentrate Hopper	

17.5.3 Kamoa-Kakula Slag Concentrator Design and Process Description

A block flow diagram of the Kamoa-Kakula slag concentrator is shown in Figure 17.5.

Figure 17.5 Kamoa-Kakula Slag Concentrator Block Flow Diagram



17.5.3.1 Slag Slow Cooling and Handling

A slag ladle carrier will be used to transport molten slag from the slag cleaning furnace to a cooling yard with 204 locations in which slag ladles can be placed for slow cooling. This will take place by air cooling over 8-hours followed by water cooling for 56-hours. Frozen slag will be tipped out of each ladle after this, onto the slag stockpile using the ladle carrier.

Spray water will be collected in a drainage ditch at the cooling yard and is then pumped from a collection pond to the water-cooling system.

The fenced slag cooling area will be approximately 6,490 square metres in area, with a concrete base.

17.5.3.2 Slag Primary Crushing

Mobile, hydraulic hammers will be used to pre-crush slag to <450 mm. A primary crusher will then be utilised to crush 117 tph of slag to P_{80} of 100 mm, after which it will be conveyed to the crushed slag stockpile. Apron feeders beneath this stockpile will be able to process 88 tph of crushed slag each, feeding a belt conveyor able to transport 140 tph of slag.

17.5.3.3 Slag Milling

A 5-metre diameter by 5-metre length SAG mill with an 1,800 kW motor has been specified, equipped with mill relining and ball loading machines. The mill will be fed from the crushed slag stockpile by a 121 tph belt conveyor. The SAG mill will be fitted with a trommel, from which 17 tph of oversized pebbles will be conveyed back to the SAG mill inlet. Trommel undersize will report to the next stage of milling.

A 5-metre diameter by 8-metre length ball mill, with a 3,000 kW motor, will follow the SAG mill, with discharge from both mills being in closed circuit with two stages of hydro cyclones to achieve controlled size classification. Cyclone underflow from both stages is returned to the ball mill, secondary cyclone overflow grading 80% <45 micron will report to flotation. First cleaner tailings and scavenger concentrate from flotation will be returned to the ball mill circuit.

The SAG and ball mills will use common ball handling and mill relining equipment.

17.5.3.4 Slag Flotation

Cyclone overflow will gravitate to the 50 cubic metre rougher flotation feed conditioning tank, in which flotation conditioning and sulfidisation is intended to take place, with the addition of multiple reagents.

Pulp will then flow to the first 40 cubic metre rougher tank cell. There will be four rougher cells, four scavenger cells, and seven cleaners. However, the cleaners are arranged in a cleaner/recleaner/final cleaner configuration so there are only two or three cells per cleaning stage.

Primary rougher concentrate will report to the final concentrate hopper. Secondary rougher concentrate will be transferred to cleaning, in three 16 cubic metre cells. Scavenger concentrate will be routed to the first cleaner cell or can be recycled to milling. Similarly, the first cleaner tail will be recycled to the second rougher cell, or to milling.

The flotation system is equipped with slag feed, concentrate and tailings samplers and XRF analysers to determine the material copper grades and assist with process control. Blowers are specified to provide forced aeration in flotation.

17.5.3.5 Slag Flotation Reagents

Reagents used in the SFP include Z-200 collector, MIBC frother, Na_2S to sulfidise the slag and pH modifier $NaHCO_3$. Generally, these are prepared in agitated tanks and then pumped to the flotation circuit reagent dosing points.

Forced area ventilation as well as ventilation and dust/vapour collection systems for certain reagents are included in the design. Reagent storage sufficient for three months has been specified.

17.5.3.6 Slag Flotation Tailings Handling

100 m^3/h of flotation tailings will be pumped to the 24-metre diameter slag flotation tailings thickener. After thickening, the overflow will be routed to the SFP return water system as process water.

Thickener underflow will be pumped to the tailings tank of the main Kamoa ore flotation concentrator. This is located about 1,400 m away.

17.5.3.7 Slag Flotation Concentrate Handling

30 m^3/h of final cleaner and primary rougher concentrate will be combined, thickened in a 12-metre diameter thickener, pumped to a filter press feed tank and filtered in a pressure filter. Thickener overflow and filtrate from the concentrate filter will be pumped to the slag flotation process water circuit.

Slag concentrates filter cake containing less than 10% moisture will be transferred by truck to the main ore concentrator wet concentrate storage area. Filter cake will be stored in a bunker with 14-day capacity.

17.5.3.8 Slag Flotation Process Utilities

The slag cooling water circulation system requires a flowrate of 490 m^3/h of water. Rainwater is expected to be sufficient to supply the system with only minimal make-up from other sources.

Process water consumption in slag milling, flotation and thickening circuits will be about 150 m^3/hour, including small amounts of gland seal water. 76.2 m^3/h of cooling water will be needed for the slag SAG and ball mills.

17.5.4 Kamoa-Kakula Slag Concentrator Services Requirements

Table 17.8 Projected Slag Concentrator Water, Power, and Consumables

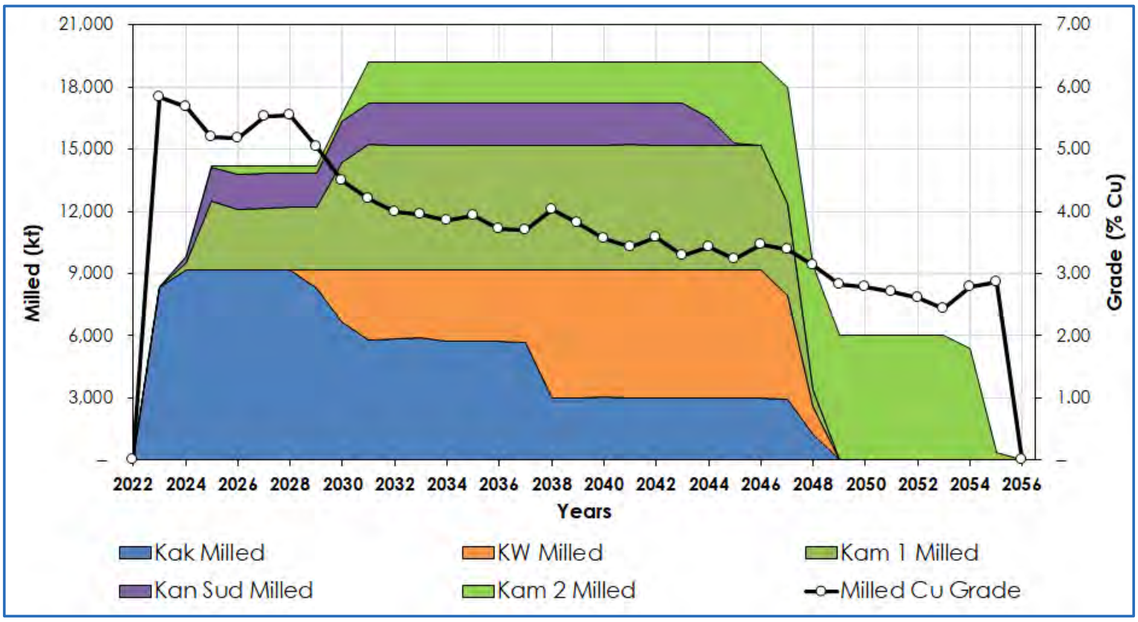
Unit name	Unit	Quantity (annual)
Auxiliary materials		
Lines	t/a	194.7
Belt	t/a	833.6
Steel balls	t/a	444.6
Z-200	t/a	55.6
Impeller cover plate	t/a	111.4
Engine oil	t/a	27.8
Grease	t/a	52.2
Pump wearing parts	t/a	5.6
Filter cloth	t/a	1,667.1
MIBC	t/a	72.2
Na_2S	t/a	138.9
$NaHCO_3$	t/a	333.4
Fuel	t/a	0
Electric power	MWh/a	34,369
Plant water	Thousand m^3/a	126.5

17.6 Kamoa-Kakula 2023 PFS Processing Production Schedule

The processing production schedule is shown in Table 17.9.

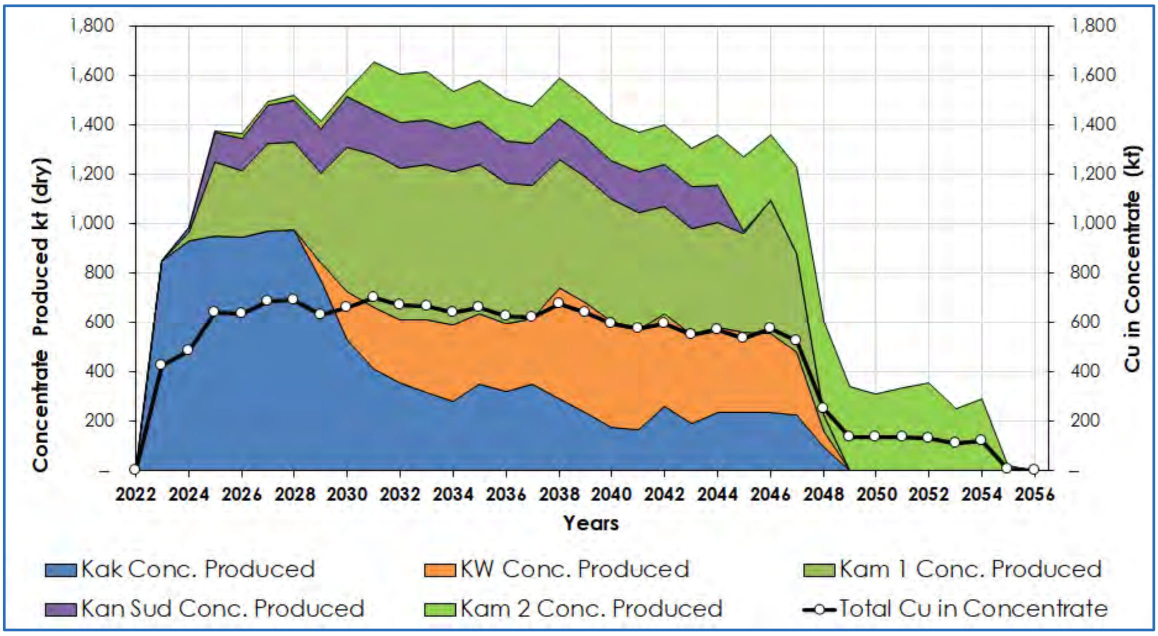
The life-of-mine processing schedule is shown graphically in Figure 17.6 and in Figure 17.7.

Figure 17.6 Kamoa-Kakula 2023 PFS Processing Schedule



OreWin, 2023.

Figure 17.7 Kamoa-Kakula 2023 PFS Processing Schedule – Concentrate Produced



OreWin, 2023

Table 17.9 Kamoa-Kakula 2023 PFS Processing Production Schedule

Description	Unit	Total	Project Time (Years)														
			2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037
Quantity Milled	kt	476,195	8,375	9,836	14,201	14,161	14,202	14,205	14,201	16,701	19,200	19,200	19,200	19,200	19,200	19,200	19,200
Cu Feed Grade	% Cu	3.94	5.83	5.68	5.20	5.17	5.52	5.54	5.05	4.49	4.21	3.99	3.95	3.85	3.93	3.72	3.71
Copper Conc. Produced	kt (dry)	37,802	850	983	1,374	1,362	1,492	1,516	1,412	1,541	1,652	1,604	1,611	1,535	1,578	1,502	1,472
Copper Concentrate Recovery	%	86.62	87.03	86.91	86.44	86.43	87.08	87.48	87.63	87.69	86.44	87.05	87.73	86.95	87.41	87.40	86.96
Copper Concentrate Grade	% Cu	43.05	50.00	49.39	46.42	46.48	45.78	45.38	44.49	42.72	42.26	41.63	41.29	41.84	41.79	41.52	42.06
Total Recovered Copper Production	Mlb	35,394	933	1,066	1,386	1,375	1,484	1,496	1,363	1,430	1,518	1,451	1,446	1,395	1,433	1,354	1,344
Total Recovered Copper Production	kt	16,054	423	483	628	624	673	679	618	649	689	658	656	633	650	614	610

Description	Unit	Project Time (Years)															
		2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050	2051	2052	2053
Quantity Milled	kt	19,200	19,200	19,200	19,200	19,200	19,200	19,200	19,200	19,200	17,977	9,390	6,003	6,004	6,004	6,004	6,003
Cu Feed Grade	% Cu	4.03	3.81	3.56	3.43	3.58	3.30	3.42	3.22	3.47	3.40	3.13	2.82	2.79	2.71	2.62	2.45
Copper Conc. Produced	kt (dry)	1,588	1,510	1,416	1,371	1,401	1,305	1,361	1,268	1,361	1,228	605	342	310	333	357	252
Copper Concentrate Recovery	%	87.36	87.22	87.05	86.97	86.67	86.57	86.93	86.58	86.27	85.63	85.04	81.61	81.02	82.31	84.51	74.66
Copper Concentrate Grade	% Cu	42.54	42.28	41.98	41.82	42.52	42.02	41.96	42.25	42.21	42.58	41.34	40.48	43.77	40.24	37.26	43.47
Total Recovered Copper Production	Mlb	1,468	1,387	1,289	1,243	1,292	1,190	1,238	1,163	1,245	1,135	577	301	295	292	290	238
Total Recovered Copper Production	kt	666	629	585	564	586	540	561	527	565	515	262	137	134	133	131	108

Description	Unit	Project Time (Years)															
		2054	2055	2056	2057	2058	2059	2060	2061	2062	2063	2064	2065	2066	2067	2068	2069
Quantity Milled	kt	5,363	365	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Cu Feed Grade	% Cu	2.78	2.86	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Copper Conc. Produced	kt (dry)	291	19	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Copper Concentrate Recovery	%	79.75	79.98	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Copper Concentrate Grade	% Cu	40.81	44.58	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Total Recovered Copper Production	Mlb	259	18	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Total Recovered Copper Production	kt	117	8	–	–	–	–	–	–	–	–	–	–	–	–	–	–

17.7 Comments on Section 17

17.7.1 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 top size 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 top size 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.

17.7.2 Smelter

Solar power with battery storage may be an alternative to emergency diesel generators that is worth considering.

Use of burnt lime flux in the smelter instead of limestone will reduce power consumption. However, the hazards to operation and maintenance personnel of using burnt lime, and the potential consequences of the marked exothermic reaction if the material is wettened must be considered carefully in the detailed design and operating practices for to be developed for the smelter.

Vendor project references for the combination of diesel burners and supplementary pulverised coal in a controlled flash smelting, or DBF operation, will provide more confidence if they can be obtained.

Granulation of SCF slag before re-treatment by flotation could be considered as an alternative to the proposed slow cooling and crushing arrangement, which requires several hundred slag ladles. Comparative comminution and flotation testwork should be carried out to support the final design solution. The mass balance, as well as the number of equipment items in slag flotation and product handling, should be investigated further to ensure that product grade objectives, recovery targets and plant availability will all be achieved.

18 PROJECT INFRASTRUCTURE

18.1 Kamoa-Kakula General Site Infrastructure

18.1.1 Introduction

Kamoa-Kakula general site infrastructure required to support mining and processing, such as bulk supply infrastructure (including water and power), waste management, buildings (Workshops, offices and stores), tailings storage facility (TSF), and main access roads.

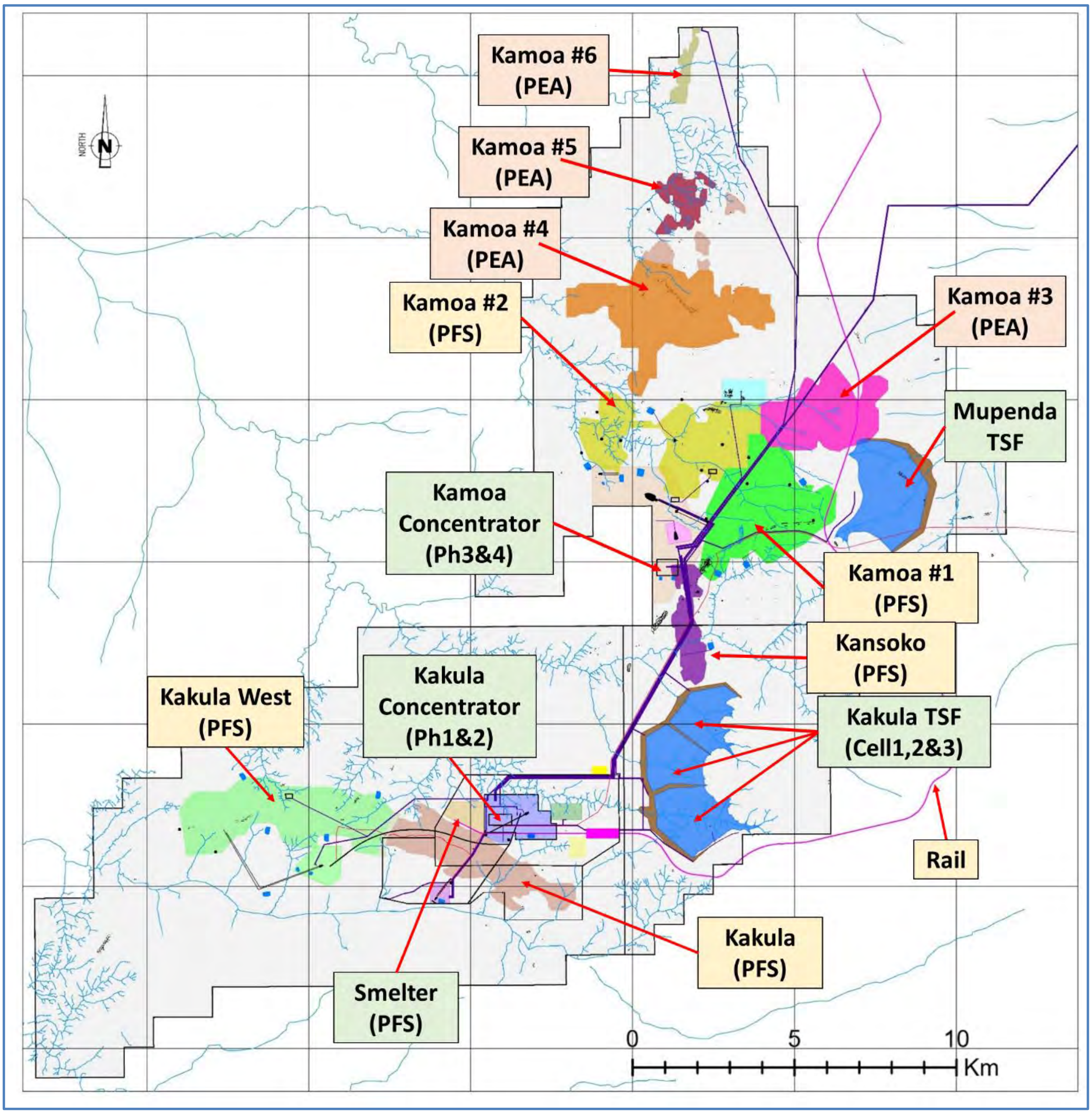
The following companies provided input to applicable facility designs:

- Kamoa Copper SA: Project team and other consultants.
- Golder WSP (Golder): Water and waste studies.
- Paterson & Cooke: Backfill.
- Knight Piésold: Surface geotechnical study.
- Epoch Resources: Tailings facility design.
- Kamoa Power: Bulk power.

18.1.2 Site Plan and Layout

A plan showing the locations of the mines and key infrastructure for Kamoa-Kakula PFS is shown in Figure 18.1.

Figure 18.1 Kamoa-Kakula 2023 PFS Site Plan

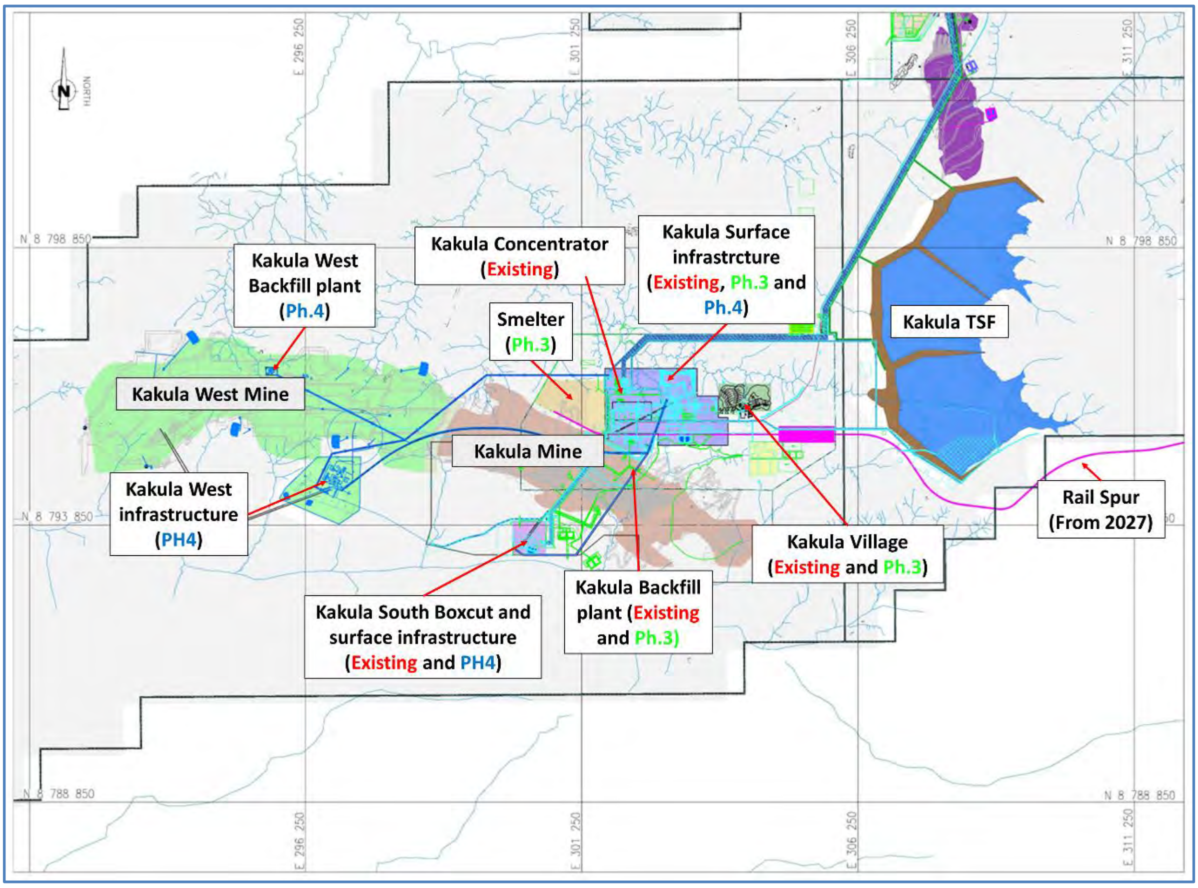


Kamoa Copper SA, 2023.

18.1.2.1 Site Plan and Layout – Kakula/Kakula West

A plan showing the locations of the mines and key infrastructure for Kakula and Kakula West is shown in Figure 18.2.

Figure 18.2 Site Infrastructure Layout Plan – Kakula/Kakula West



DRA, 2023.

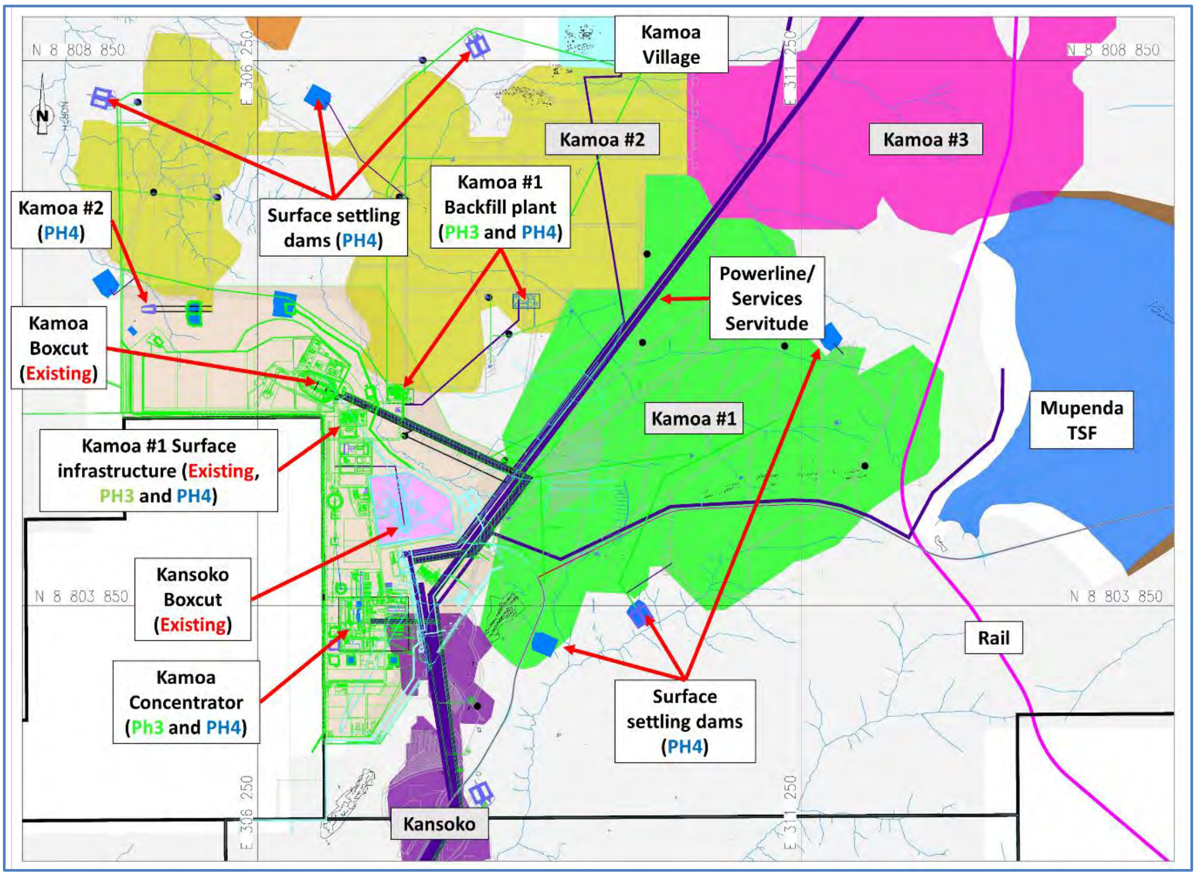
Kakula expansion will include the construction of a new overland conveyor from Kakula South box-cut. The smelter, railway spur, upgrade of the backfill plant are all part of the Kakula expansion.

Kakula West includes a new surface conveyor to the Kakula process plant and all associated surface infrastructure required to bring Kakula West into full production.

18.1.2.2 Site Plan and Layout - Kamoa

A plan showing the locations of the mines and key infrastructure for Kamoa is shown in Figure 18.3.

Figure 18.3 Site Infrastructure Layout Plan - Kamoa



DRA, 2023.

The existing infrastructure around the Kansoko mine mainly consists of surface dewatering facilities, the Kansoko box-cut, and temporary offices and stores. Phase 3 involves the building of the Kamoa surface infrastructure, a concentrator plant, and a bulk power supply. In Phase 4, sustaining capital is also included, which is necessary to maintain a production rate of 10-million tonnes per year at Kamoa.

Construction of the Mupenda TSF will commence after the completion of the Kakula TSF Cells 1, 2, and 3. The tailings from the Kamoa concentrator will be pumped to the Kakula TSF Cells 1, 2, and 3 when necessary. Additionally, a return water pipeline from the Kakula TSF will be constructed to the Kamoa plant.

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.

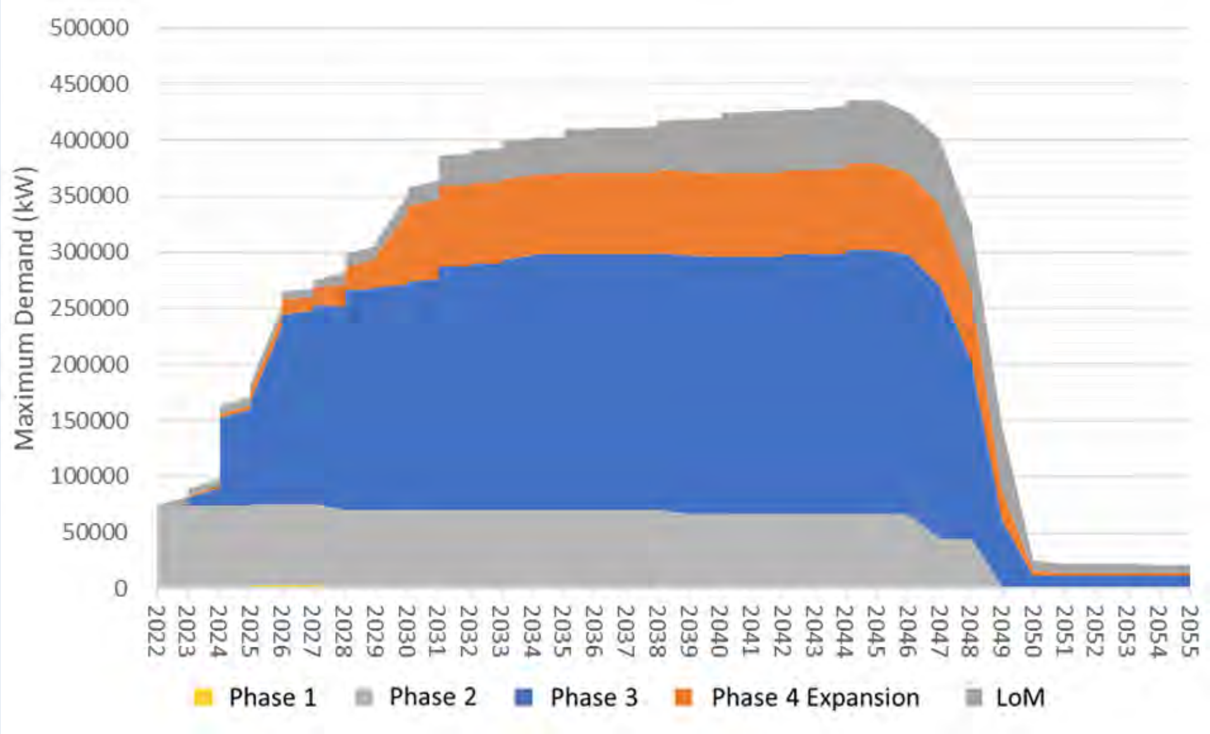
The site layout was designed to accommodate the Phase 3 and Phase 4 expansion which is added to the current execution block plan master file to ensure the long-term planning requirements are considered during the construction of Phase 3. Special future considerations are required on the surface conveyors, between Kakula West and Kakula South box-cuts to the Kakula processing plant. Surface conveyor from Kansoko box-cut tie-in to the Kamoa processing plant.

18.1.4 Power Supply

18.1.4.1 Estimated Electrical Consumption and Maximum Demand

A bottom-up estimation methodology was used to predict electrical power consumption, and the Maximum Demand (MD), for the proposed surface and underground installations. The MD is the maximum electrical power demand in kVA over a 30-minute period. The load estimate was calculated by generating a load list per area in MS Excel, with all power requirements as indicated in the Mechanical Equipment List (MEL). The MEL and Process Flow Diagrams (PFDs) were used as load list inputs. The mechanical power requirements were subjected to load capacity de-rating, diversity, and utilisation factors to compensate for operating conditions and obtain a realistic value for the running power (MD). The mining/production schedules were also applied to the running loads to obtain an MD load profile in kW. The Maximum demand profile is illustrated in Figure 18.4

Figure 18.4 Maximum Power Demand by Phase



18.1.4.2 Bulk Power Supply and Transmission

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 Holdings Ltd.

Ivanhoe Mines Energy DRC recently (2021) completed the rehabilitation of six turbine generators at the Mwadingusha hydropower plant (HPP) in south-east DRC and restored the plant to its installed capacity of 78 MW during the construction, and commissioning, of the first phase of the Kamoa-Kakula Concentrator. The securing of power for the Kamoa-Kakula Project is done by Ivanhoe Mines Energy DRC on a loan agreement from Kamoa with SNEL that will be repaid on a 40% discounted consumption charge.

For the Phase 3 upgrades, the Kamoa Board has agreed to extend the loan agreement with La Société Nationale d'Électricité (SNEL), for the upgrade of unit 5 (G25) at Inga II hydropower plant (HPP) in the South-west DRC. Inga II hydropower plant is shown in Figure 18.5.

Figure 18.5 Inga II Hydropower Plant

This unit will be capable to export 178 MW over a 1,700 km, 500 kV High-Voltage Direct Current (HVDC) line to the Kolwezi area, where it is converted back into Alternating Current (AC) and tied into the 220 kV grid at the 220 kV SCK substation in Kolwezi. The SCK substation is a major 220 kV transmission station in the SNEL's southern network. Upgrading of the filter banks at Inga II, as well as at SCK, will also be part of the loan agreement, in order to boost the HVDC line's transmission capacity. Part of this project include the installation reactive power compensation equipment at SCK 220 kV substation. HVDC line from INGA to Lubumbashi is shown in Figure 18.6.

Figure 18.6 HVDC OHL from INGA to Lubumbashi

KCSA, 2023.

The upgrading is part of a programme planned, to eventually overhaul, and power boost output. On completion of the upgrading programme, a combined total of 278 MW of long-term, clean electricity will be produced for the DRC's national grid.

However, to meet the power requirements of the Phase 4 expansion, a total of 440 MW supply will be required. Refurbishing costs have been included in the study to refurbish another INGA plant turbine that will supply the Project with additional 178 MW over the same lines as described above, this will increase supply to 456 MW.

NRO to the Switch Yards of Kamoa 1 and Kakula

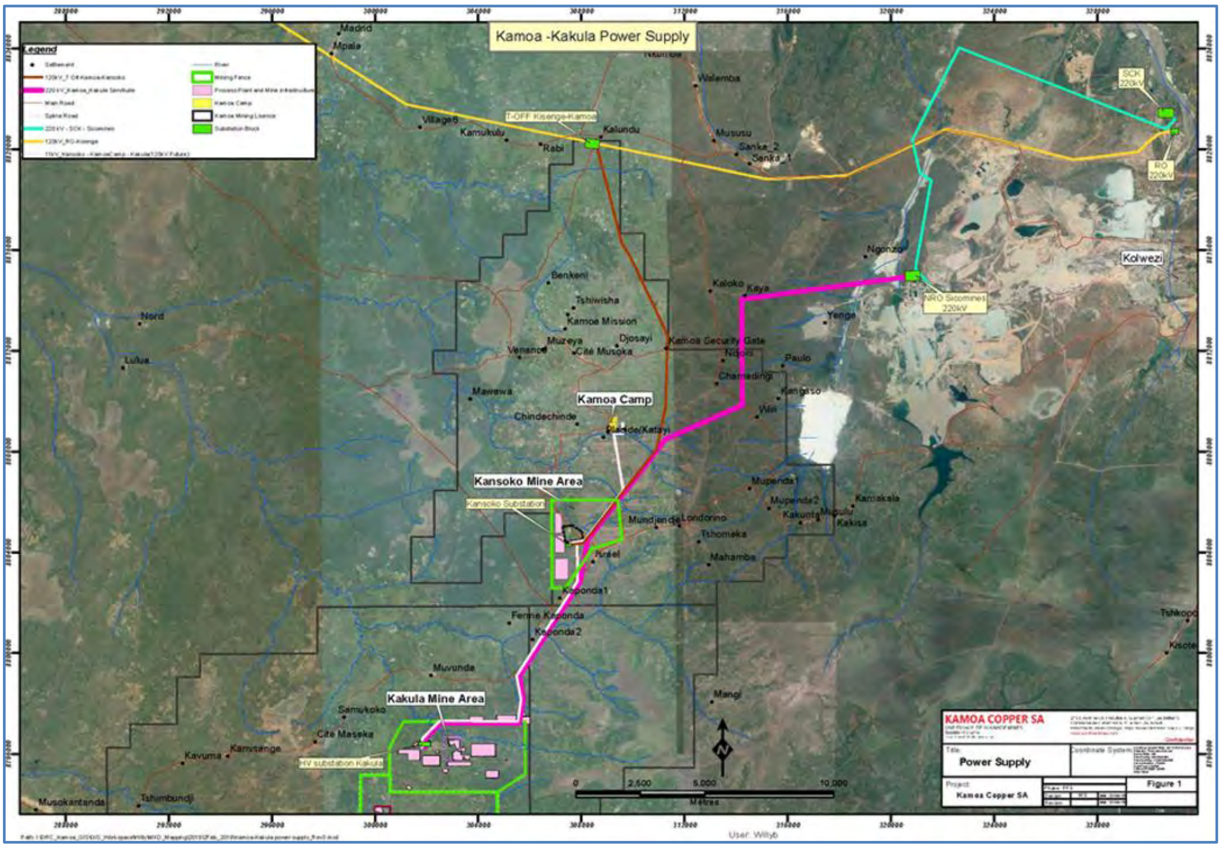
Power is currently supplied to the new Kamoa Copper Mine's 220 kV Kakula Consumer Substation (KCS) from the new SNEL substation, called Nouveau Repetiteur Ouest (NRO). The NRO substation is financed by Ivanhoe Mines Energy DRC, and forms part of the previously mentioned loan agreement. A double circuit 220 kV overhead transmission line (35 km) was installed between NRO, and the Kamoa 220 kV substation. Three 220/33 kV 80 MVA transformers were installed at the Kakula switch yard, with three more to be installed at the Kamoa 1 switch yard. The supply will conform the N+1 redundancy on the transmission line and transformers. For the smelter load requirements another two 80 MVA transformers will be installed at the existing Kakula substation.

For the phase 3 power requirements, a new 220 kV switch yard will be constructed the 220 kV Kamoa 1 substation. This yard will be centrally located close to the existing 220 kV OHL coming from NRO, towards the 220 kV KCS. The existing OHL will be diverted into the Kamoa 1 yard and will feed the 220 kV KCS yard from this substation. As such, no new 220 kV OHL will be constructed from the NRO, or Kamoa 1 switch yards.

Metering will be done at the take-off point at the NRO substation, as the overhead line, and two switch yards (Kakula Consumer Substation and Kamoa 1 Consumer Substation) are the property of Ivanhoe Mines Energy DRC.

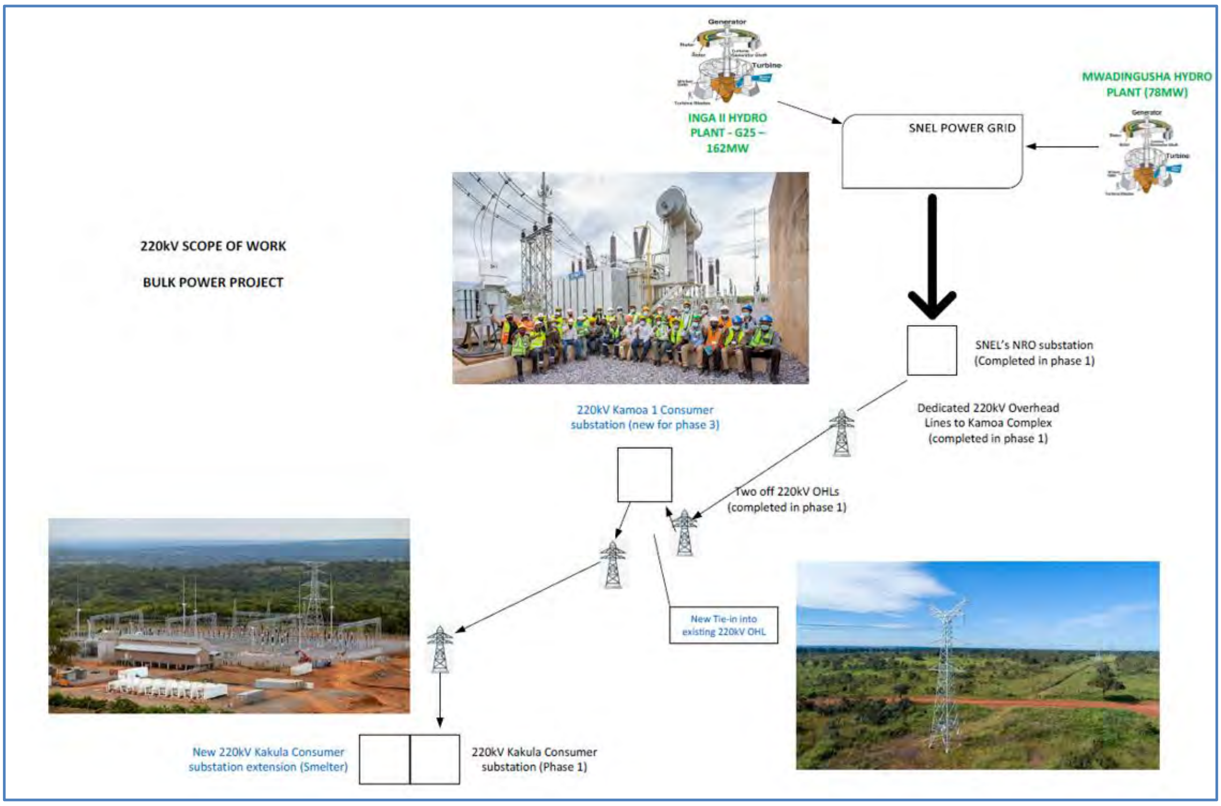
The purple line in Figure 18.7 indicates the 220 kV line to Kamoa, and the blue line indicates an existing OHL supplying Sicomines mine.

Figure 18.7 Bulk Power OHL from Kolwezi's NRO



See Figure 18.8 for a simplified representation of the 220 kV reticulation.

Figure 18.8 High-Level Simplified Representation of 220 kV Reticulation



18.1.4.3 Construction Power Supply

Power supply for the construction period will be sourced from the 11 kV Infrastructure network at Kansoko. Construction power will be provided at the following areas: Kamoa #1 Red-zone, Kamoa #1 Process Plant Dry circuit, and Kamoa #1 Process Plant Wet circuit.

18.1.5 Electrical, Control and Instrumentation Design

18.1.5.1 Design Basis

The electrical design is based on equipment specifications and electrical design criteria. The electrical equipment is designed or selected to:

- Provide for high plant availability.
- Ensure only proven technology is used.
- Provide an effective, simple solution that plant operating personnel can maintain.
- Provide a safe working environment for personnel and equipment.

Every effort has been made to increase energy efficiency and reduce adverse environmental effects.

18.1.5.2 Voltage Selection

As per the electrical design criteria, the selected voltages for the project are as follows:

- Medium voltage systems:
  - Distribution voltage surface: 33 kV
  - Distribution voltage surface and underground: 11 kV resistively earthed.
  - Nominal frequency: 50 Hz
- Low voltage systems:
  - Mine stoping and developing equipment operating voltage: 1,000 V AC resistively earthed.
  - Mine UG Infrastructure: 690 V AC resistively earthed+
  - Concentrator, backfill plant and surface infrastructure operating voltage: 690 V AC resistively earthed.
  - Infrastructure operating voltage: 690 V AC resistively earthed.
  - MCC control voltage: 110 V AC solidly earthed.
  - Small power LV voltage: 400/230 V AC

18.1.5.3 Power Factor Correction

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 will be determined during the execution phase.

18.1.5.4 MV Distribution

220/33 kV Kakula KCS Substation

Bulk 33 kV power for the mining and plant operations for the Kakula operations will be provided from Kakula's 33 kV Central substation. All bulk power will be distributed via dual supply to ensure N+1 redundancy. PFC will only be installed at MV level where required.

Power Distribution – Concentrator Plant

The Concentrator plant MV distribution will be done from a dedicated plant 33 kV MV substation supplied from 33 kV Kakula KCS substation. From this substation there will be redundant feeds to the following plant sections:

- Crushing and screening
- Milling
- HPGR

- Concentrator
- Filtering
- Common services

33 kV Power Distribution – Kakula West

New dedicated 33 kV overhead powerlines will be installed from the 33 kV Kakula KCS substation to feed the Kakula West portal substation. The portal substation will be equipped to MV switchgear and 33/11 kV transformers to feed the 33 kV ventilation substations, 33 kV refrigeration substations, 33 kV backfill plant substations and 11 kV underground sections via medium voltage cables. 30 MV 33 kV/11 kV transformers and 33 kV/690 V mini-substations will power the box-cut area.

220/33 kV Kamoa KCS Substation

Infrastructure will be supplied from the 33 kV Kansoko KCS substation installed at Kansoko (adjacent to the 220 kV yard). 33 kV power distribution will be done via overhead line and cables in a N+1 redundant format to ensure supply reliability.

Power Distribution – Kamoa Concentrator Plant

The Kamoa Concentrator plant MV distribution will be done from a dedicated plant 33 kV MV substation supplied from 33 kV Kamoa KCS substation. From this substation there will be redundant feeds to the following plant sections:

- Crushing and screening
- Milling
- HPGR
- Concentrator
- Filtering
- Common services

33 kV Power Distribution – Kansoko Sud

The Kansoko Sud 33/11 kV substation will be supplied from the 33 kV Kamoa KCS substation via MV cable. The design allows for an N-1 redundancy to ensure supply reliability.

33 kV Power Distribution – Kamoa 1

Power is distributed to Kamoa 1 via two 100 MVA rated OHLs. The design allows for an N-1 redundancy to ensure supply reliability. A 33 kV Kamoa 1 Portal substation is planned.

33 kV Power Distribution – Kamoa 2

Power is distributed to Kamoa 2 via two 50 MVA rated OHLs. The design allows for an N-1 redundancy to ensure supply reliability. A 33 kV Kamoa 2 Portal substation is planned for when mining will commence through the new Kamoa #2 box-cut, only much later in mine life.

18.1.5.5 Generators

MV generators 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 generators, switch off non-essential breakers and optimise the generator power plant's efficiency. The following generator installations have been allowed for:

- 10 x 3.2 MW Generators at Kamoa 1
- 5 x 3.2 MW Generators at Kamoa 2
- 10 x 3.2 MW Generators at Kakula West

In addition, a 40 MW back-up generator will be constructed that will serve as back-up power to keep some operations going during power outage.

18.1.5.6 LV Distribution

The MCC distribution voltage will be 690 V for surface and underground infrastructure. Suitably sized transformers, i.e., 2,500 kVA 33 kV/690 V and 11 kV/690 V Dyn11 ONAN step-down transformers will be installed. Transformer neutral points will be resistively earthed. These transformers will power the MCCs.

18.1.5.7 Lighting

Non-essential lighting and small power supply in each plant area will be taken from independent sub-boards supplied from 33 kV/400 V and 11 kV/400 V Dyn11 ONAN transformers and mini substations. These include offices, workshops, change houses and other 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.

18.1.5.8 Control Systems

The control system architecture is designed around a PLC and central Supervisory Control and Data Acquisition System (SCADA). The Mining Control Rooms (MCRs) are located on surface near the main decline area for the control of daily mining operations, on surface and underground. The 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 relevant points will take place over optic fibre using 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.

18.1.5.9 Instrumentation

The instrumentation system design is based on equipment specifications and Control and Instrumentation (C&I) design criteria developed for the project. In general, except for belt-scales, and density metres that communicate via a ProfiNet (PN) fieldbus, conventional "hard-wired" type instrumentation is used in the design. Instrumentation is based on standard signal types and will be wired directly to weatherproof, field-mounted RIO boxes, located strategically around the plant. RIO boxes will be connected on a copper or fibre link back to the relevant control room.

18.1.6 Backfill Plant

The backfill plant locations were determined such that the various orebodies could be supplied with backfill, considering the mining and ramp-up rates. Tailings will be supplied to the backfill plant from the concentrator plant/s. Backfill tailings supply lines are included in the concentrator plant scope, and the various distances between the concentrator/s and backfill plant/s were taken into consideration.

Kakula mine will be supplied by the existing Kakula backfill plant that comprises three modules in total. The Kakula West orebody can be supplied from a single location by a plant that will comprise three backfill modules. The Kansoko, and Kamoa mines, required two separate plant locations due to the extent of the mines, with each plant comprising of two modules.

The process for preparing the backfill is as per the existing backfill plant located at Kakula. The tailings feed will be stored in filter feed tanks at the paste plant. From there, the tailings are pumped to vacuum disc filters, with the filter cake reporting to the continuous mixer. The continuous paste mixer receives filter cake, trim slurry, water, and cement from the binder system. The continuous paste mixer will discharge paste into a paste hopper to provide a buffer for the paste pumping system. From the hopper a positive displacement pump will transfer the paste via the surface piping and boreholes to the underground deposition points.

18.1.7 Kakula Tailings Storage Facility

Epoch Resources (Pty) Ltd (Epoch) completed the designs of the TSFs as part of the Kamoa-Kakula 2023 PFS. 292 million tonnes of tailings will be produced by the mine, therefore two TSF sites will be developed, namely Kakula TSF and Mupenda TSF. The Kakula TSF will comprise 3 Cells. The TSFs will be raised as downstream facilities, comprising several raises of the embankment, sourced from local borrow areas.

The preferred sites were based on-site selection studies completed during various conceptual and feasibility studies since 2014. The Kakula TSF site was selected as the first TSF to be constructed given its proximity to the Kakula mine and concentrator, as well as the expected low consequence classification.

18.1.7.1 Design Criteria

The Kakula TSFs have been designed with a LOM tailings production of 292 Mt over 33-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 18.1.

Table 18.1 Design Criteria

Item	Criteria	Value	Source
1	Design life of facility	33-years	Kamoa
2	Total Tailings	292 Mt	Kamoa
2.1	Kakula	137.6 Mt	Kamoa
2.2	Kamoa	140.0 Mt	Kamoa
2.3	Smelter Tailings	14.6 Mt	Kamoa
3	Particle Specific Gravity	2.923	Kamoa/ST Lab
4	Average Dry Density	1.28 t/m^3	ST Lab/Epoch
5	Average Particle Size Distribution	75% Passing 60 µm	ST Lab
6	% Solids to Water (by Mass)	48	DRA/Kamoa
7	Delivery Method	Hydraulically pumped	Kamoa
8	Geochemistry	Leachable Mine Waste	Golder

18.1.7.2 TSF Site Selection

A TSF site selection study was undertaken in 2017. The required capacity for the study was for 228 million tonnes, to accommodate possible further expansion. The preferred site was found to be the Kakula TSF site. The available capacity at the Kakula site, subsequently reduced to 197 million tonnes, due the reduced density following testing of a tailings sample.

The second site selection study was undertaken for a separate TSF for Kamoa. The two preferred sites for this study were identified as the Mupenda site, and TSF 8 site. Mupenda was found to be the preferred option of the two, given the contaminated catchment downstream, while TSF 8 is located in a pristine catchment.

Subsequently, it was found to be feasible to send tailings, from both Kakula and Kamoa, to a single TSF, thereby delaying the costs to construct Mupenda. This has increased the rate of construction of the TSFs in order to provide the required capacity.

18.1.7.3 Geotechnical Investigation

A geotechnical investigation of the Kakula TSF cell 1 site was undertaken by Knight Piésold. This included profiling of test pits and boreholes, sampling of soils, and laboratory testwork 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.

18.1.7.4 Seepage and Stability Assessment

Stability analyses were carried out as part of the Feasibility Study of Kakula TSF Cell 1. 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 TSF.

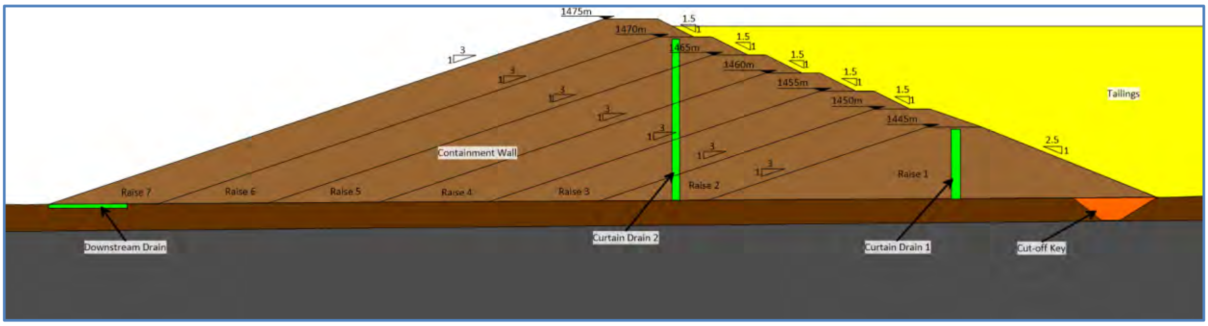
18.1.7.5 Operational Methodology

The depositional technique selected for this project will be a valley containment, hydraulically deposited, spigot facility. The containment embankment wall will be constructed using borrow material and tailings will be deposited behind the wall embankment and into the valley. This design is a common construction technique used in tailings storage facilities.

The Kamoa-Kakula Project will comprise two TSFs, namely the Kakula and Mupenda TSFs. The Kakula TSF will consist of three compartments, or cells, over three valleys. Each cell will be phased over a number of raises of the embankment which will progress as a downstream facility in 5 m raises (Figure 18.9), therefore relying on the construction of an engineered, earth embankment to store tailings. The southernmost compartment (cell 1) will be constructed first and comprises seven wall raises. The middle and northernmost cells will have four and three raises, respectively.

The Mupenda TSF will be constructed after the Kakula TSF is completed and will comprise four wall raises, constructed in the same manner as the Kakula TSF.

Figure 18.9 Typical Phasing of the TSF Embankment



Epoch, 2022.

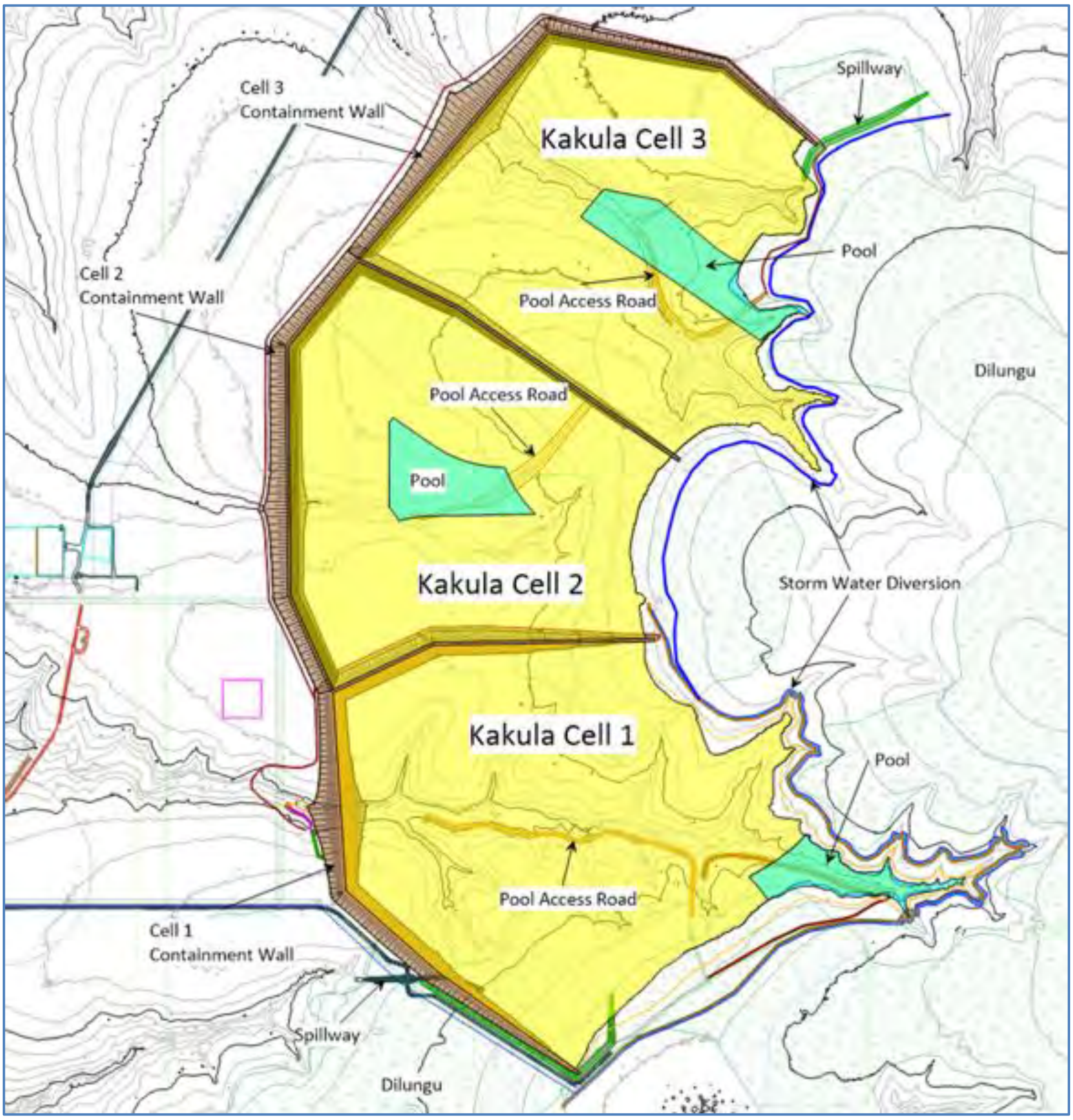
The Kakula TSF Cell 1 started receiving tailings from Kakula in May 2021 and will start receiving tailings from Kamoa in 2024. Cell 1 will reach capacity in 2031, at which point Cell 2 would be ready to accept tailings. Cell 2 and Cell 3 will reach capacity in 2036 and 2041, respectively. The construction of the raises will be continuous for the life-of-mine in order to provide the required storage capacity.

18.1.7.6 Key Design Features

The layouts of the TSFs are shown in Figure 18.10 and Figure 18.11 and the key design features of the facility are as follows:

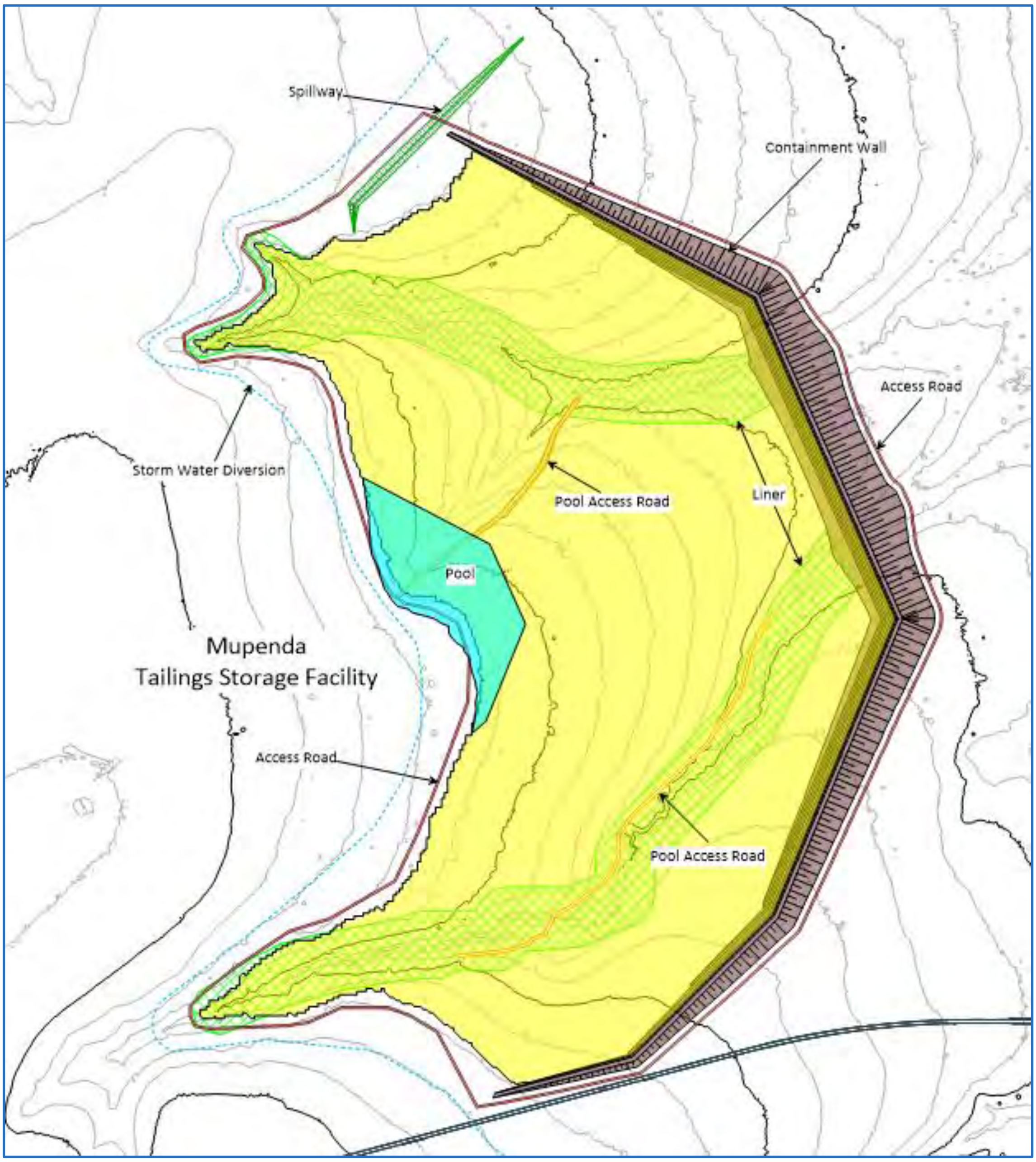
- Full containment, downstream construction method, with open ended deposition.
- An engineered, earth fill embankment with an HDPE liner on the upstream slope.
- Pool access road.
- A floating decant system.
- Curtain and blanket drain seepage collection systems inside the containment wall (to control the phreatic level within the wall).
- An HDPE liner (over highly permeable soils) and associated seepage cut-off drains.
- Storm water diversion trenches.
- Emergency spillways.

Figure 18.10 Kakula TSF Layout



Epoch, 2022.

Figure 18.11 Mupenda TSF Layout



Epoch, 2022.

18.1.7.7 Risk Identification

The major risks associated with the TSF are as follows:

- The preferred site lies upstream of 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.
- Suitable borrow areas close to the TSF site have not yet been identified, and material may need to be sourced from further away resulting in delays and increased costs.
- 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, 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 extreme rainfall events. Failure to provide sufficient storage may lead to water overtopping the embankment. The storm water diversions must be maintained to ensure that minimal run-off enters the TSFs.
- Tailings elevation inside the TSF must be monitored to ensure that the construction of the wall raises is scheduled correctly to ensure tailings freeboard in maintained and capacity is available.
- Further geotechnical data will be required for TSFs after Kakula TSF cell 1 as this information is critical to ensure the stability of the TSF has been addressed.

18.1.8 Bulk Earthworks

18.1.8.1 Geotechnical Investigation

Knight Piésold (Pty) Ltd. (KP) was appointed by DRA on behalf of Kamoa Copper SA to conduct a geotechnical investigation for the Phase 3 development of the mine. This includes the Kakula Mine smelter, environmental control dams and Kamoa 1 (previously Kansoko). Their report provides the geotechnical investigation results for the Kamoa 1 site. The geotechnical investigation for the concentrator, plant infrastructure and other ancillary mining structures for Kamoa 1 is completed.

The KP report documents the fieldwork (test pitting and laboratory testing) results conducted at the Kamoa 1 site between 11 February, and 13 May 2022. This includes additional test pitting conducted to further investigate potential borrow areas from 5 to 13 May 2022.

The investigation aimed to determine the ground conditions in the planned area of investigation and provide recommendations for founding and re-use of in situ materials for construction. These considerations were incorporated in earthwork designs and costing.

18.1.8.2 Foundation Recommendations

Light Structures (50 To 150 kPa Bearing Pressure)

The following earthworks are recommended for light structures:

- Strip topsoil to a depth of 0.5 m and stockpile for future use or use for landscaping.
- Excavate soil which mostly comprises hill wash (<G9 quality) to a depth of 2.5 m below the ground surface and remove to spoil. For strip and lightly loaded pad foundations, the excavation depth can be shallower and needs to extend to a depth of 1.5xB where B is the foundation width. The excavation must extend to 1.5x the foundation width beyond the footprint of the foundations. The excavation walls must not be vertical and should be battered to an average 600 slope (i.e., 1V:0.6H) for short-term stability. Large excavations may be considered if some of the smaller excavations for individual structures are combined.
- Compact excavation floors to 90% of Modified AASHTO Maximum Dry Density (MDD) at Optimum Moisture Content (OMC).
- Backfill the excavations with G7 material and compact in maximum 200 mm thick layers to 95% of modified AASHTO MDD at OMC up to ground level or above ground level to construct engineered platforms for the plant structures. Excavate the G7 backfill, then construct the foundations. For large foundations the G7 material can be backfilled up to the foundation founding level and foundations can be constructed directly on the backfill/engineered soil raft. G7 material can then be used to backfill around the foundations up to ground level. The maximum allowable foundation bearing pressures may not exceed 150 kPa.
- Aprons should be constructed around all structures to mitigate water ingress below the foundations. The platforms must be sloped for drainage of surface water runoff.

The following structures are typically classified as light structures:

- Office buildings,
- Conveyors,
- Workshops,
- Stores,
- Open storage areas, and
- Substations.

Moderately Heavy Structures (150–250 kPa Bearing Pressure)

The following earthworks are recommended for moderately heavy structures:

- Strip topsoil to a depth of 0.5 m and stockpile for future use or use for landscaping.
- Excavate soil which mostly comprises hill wash (<G9 quality) to a depth of 3 m below the ground surface and remove to spoil. For smaller foundations the excavation depth can be shallower and needs to extend to a depth of 1.5xB where B is the foundation width. The excavation must extend 1.5x the width of the foundation m beyond the footprint of the foundations. The excavation walls must not be vertical and should be battered to an average 600 slope (i.e., 1V:0.6H) for short-term stability. Large excavations may be considered if some of the smaller excavations for individual structures are combined.
- Compact the excavation floors to 90% of Modified AASHTO MDD at OMC.
- Backfill the excavations with G6 material and compact in maximum 200 mm thick layers to 95% of Modified AASHTO MDD at OMC up to ground level or above ground level to construct engineered platforms for the plant structures. Excavate the G6 backfill and then construct the foundations. For large foundations the G6 material can be backfilled up to the foundation founding level and foundations can be constructed directly on the backfill/engineered soil raft. G6 material can then be used to backfill around the foundations up to ground level. The maximum allowable foundation bearing pressures may not exceed 250 kPa.
- Alternatively, excavations can be backfilled with mine waste rock in layers not exceeding 500 mm and compacted with a large (10 tonne or 15 tonne) vibratory pad foot roller with a minimum of four passes. Rock sizes should not exceed two thirds (330 mm) of the layer thickness.
- A minimum 1.5 m thick layer of G6 material must be placed above the rockfill. The G6 must be compacted in maximum 200 mm thick layers to 95% of Modified AASHTO MDD at OMC.
- Aprons should be constructed around all structures to mitigate water ingress below the foundations. The platforms must be sloped for drainage of surface water run-off.

The following structures are typically classified as moderately heavy structures:

- Crushing and screening structures,
- Transfer structures, and
- General plant structures (except for those listed as heavy structures).

Heavy Structures (250–500 kPa Bearing Pressure)

For heavy structures, a similar approach can be followed for earthworks as was followed for moderately heavy structures. Excavations must, however, extend into soil with a very stiff consistency, which is typically from a 5 m depth below the ground surface.

The excavation can be backfilled using a combination of mine waste rock or G5 quality material compacted in 200 mm thick layers to 95% of Modified AASHTO MDD at OMC, or only G5 material to construct an engineered soil raft.

The maximum allowable foundation bearing pressures may not exceed 500 kPa. The recommended slopes for the excavation walls are 1V:2H in soft to firm soils and can be steepened to 1V:1H in stiff to very stiff soils.

Piling is not recommended for any of the structures, as excavation of unsuitable soil followed by backfilling with suitable backfill material, can be used at Kamoa 1. However, this will require deep excavations for heavy structures and the depth will vary depending on soil conditions.

The following areas will require high-specification terracing:

- Concrete stockpile tunnels,
- Mill, and
- Flotation.

18.1.8.3 Settling Ponds

Hard rock excavation is expected for the settling ponds area below an average depth of 1 m due to the presence of shallow rock. The ponds should be designed to have shallow excavation depths with an earth embankment wall constructed above ground to provide the required storage capacity.

18.1.8.4 Borrow Pits

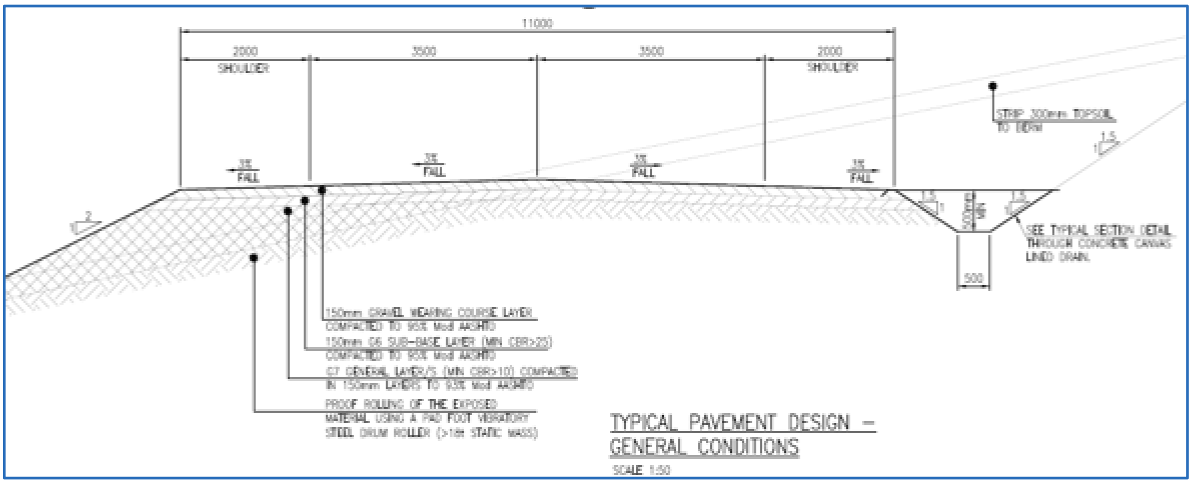
A potential borrow area has been identified in the settling pond area where shallow rock conditions occur. From the available test pit data, the average soft rock level is at 2.1 m depth over an area of about 90 Ha. A possible 1,800,000 m^3 of material of G7 to G5 material has been identified.

18.1.8.5 Roads and Parking

External Roads

External roads include the construction of main access roads (gravel) between Kamoa and Kakula, and between Kakula, and Kakula West. See Figure 18.12 for a typical cross-section. All other external roads have been completed during Phase 1 and 2.

Figure 18.12 Typical Section of Access Road



DRA, 2023.

Internal Roads and Parking

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 applicable work areas. The truck parking area at the main gate (Kakula) can accommodate 300 trucks. Internal roads and parking will consider the traffic flow inside the mine area. Security gates separate areas 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.

18.1.8.6 Material Storage Stockpiles

Both ore and waste rock stockpiling includes the stockpiling of all commodities generated during both construction, and operational phases. These commodities include:

- Low-grade (LG) ore,
- High-grade (HG) ore,
- Waste rock uncrushed and crushed, and
- Subsoils (soft and hard).

Materials deposited onto temporary stockpiles are spread further by means of mobile earthmoving equipment (trucks, loaders and dozers) to build individual type stockpiles. Stockpiles have been designed with the required footing and drainage. The footprint of each stockpile was modelled to optimise the required capacity.

A further requirement for waste rock is to re-use it as earthworks backfill material. To cater for this, waste rock must be crushed and screened to suitably sized materials. A mobile crushing and screening plant will be used for this. This plant comprises a modular jaw crusher with a vibrating grizzly, which is manually fed by a Front-End Loader (FEL). The crushed product then passes over a triple-deck screen with the undersized material being stockpiled, while the oversized material is routed to a cone crusher to be recirculated. Each mining area will have a unique stockpile layout based on the layout and space available in the area. The bulk stockpiles are used for start-up and as a strategic buffer during the LOM.

18.1.8.7 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 it directly impacts the overall capital estimate. The objective of this model is to indicate the nett quantities of material/commodities requiring stockpiling, whether temporarily or permanently. This model indicates commodity requirements and whether commodity importing is required.

The different types of earthworks commodities are:

- Topsoil
  - Topsoil is the first 200 mm to 300 mm of material removed (cut to stockpile) from on-surface excavations. Topsoil must be stockpiled in a defined (topsoil only) area.
- Subsoil (Soft Excavation)
  - Subsoil is the next type of soft soil material removed (cut to stockpile) from excavation after topsoil removal. The depth of subsoil in the excavation is determined by either bottom of excavation level or when refusal level (hard material) has been intersected. Subsoil could be used (in certain instances) as fill material.
- Hard Excavation
  - Hard excavation is the removal of material in the excavation that cannot be efficiently extracted or loaded by a track-type excavator. This type of excavation generally includes material like formations of unweathered rock, which can only be removed/excavated once blasted. The material from this type of excavation can also be classified as waste rock.
    - G4-G8 Material

      This material is used in layer works and complies with certain material properties and classifications. It can be obtained naturally from a borrow pit or rocky type of material that requires crushing.
    - Waste Rock

      Waste rock is material produced from underground mining activities and conveyed to surface. This material will be routed to the waste rock stockpile.

Surface Earthworks Material Flow

- The flow of surface earthworks material can be divided into two categories:
- Cut-to-Stockpile
  - Cut-to-stockpile is loading of material generated from excavations. This material is hauled, dumped and stored in a dedicated spoil or stockpile area.
- Cut-to-Fill
  - Cut-to-fill is loading of material obtained from stockpiles or commercial sources. This material is hauled to the required position, processed, levelled and compacted to the required specification.

18.1.9 Storm Water Management

Overall stormwater management, including peripheral stormwater designs were completed by Golders WSP. A stormwater management system separating contact and non-contact water was implemented at Kakula North and South Shafts, and Kakula concentrator. With the expansion of the Kamoa mining area, a stormwater management system must be implemented here too.

Golder WSP modelling was completed for Kamoa as part of the Phase 3 construction that started, and allowance was made for Kakula West appropriate to PFS level of design. Deigns and recommendations were made, 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-in-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.

Environmental protection and reduction of surface water impact will dictate these additional criteria:

- Separate non-contact and contact water – no mixing allowed between systems.
- Discharge non-contact water off-site.
- Capture contact water for re-use or management as decided.

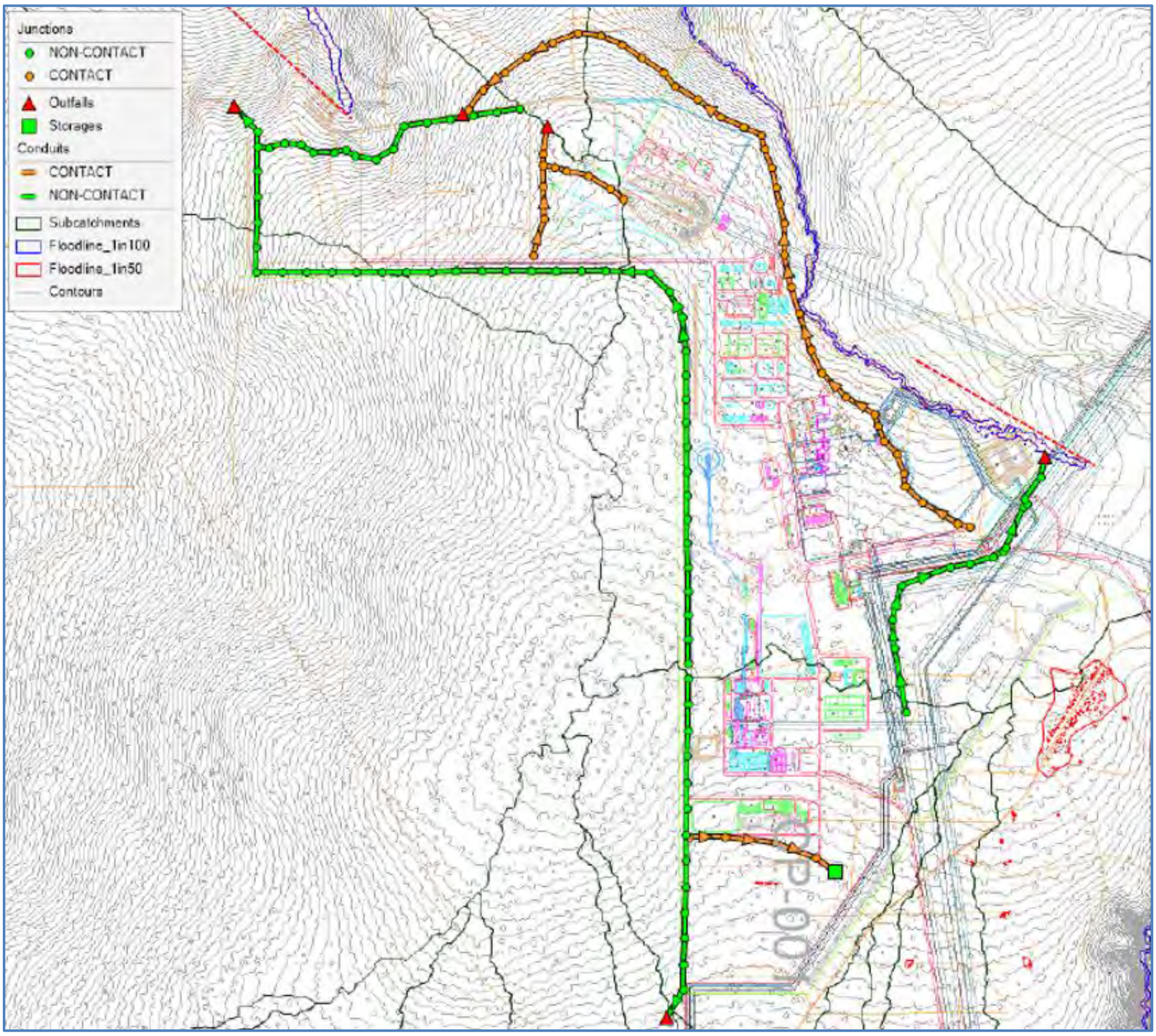
The following general considerations are presented. This is not an exhaustive list and engineering due diligence must be applied during detailed design.

- Flooding of plant areas from outside the plant footprint must be limited.

- Sediment will be mobilised during various construction and operational phases and, periodically, all channels will collect sediment. Consideration must be given to using dust suppression compounds and vegetation on sub-catchments to limit erosion and delivery of sediment to drainage systems.
- Frequent de-sedimentation of channel ponds will be imperative.
- Vegetation clearance in channels will be required.
- Channels will require scour protection (heavy-duty geosynthetic filter and riprap or gabion mattresses) and regular maintenance.
- Discharge points will require scour protection aprons, and possibly stepped structures if topography dictates.
- Larger fractions of sediments can be settled out for smaller storm events in in-line or off-channel sediment traps. For these to remain serviceable, regular removal of laid-down sediment will be necessary.
- No standing water must be allowed in channels (particularly in contact water channels). Currently, no lining materials have been specified in the storm water model, other than riprap scour protection.

Channel routes were selected to optimise the collection of non-contact and contact water. The routes present viable corridors with topographical slopes that will convey flow by gravity. See Figure 18.13 for the general arrangement of all main storm water run-off collection channels.

Figure 18.13 General Arrangement of Stormwater Model Layout



DRA, 2023.

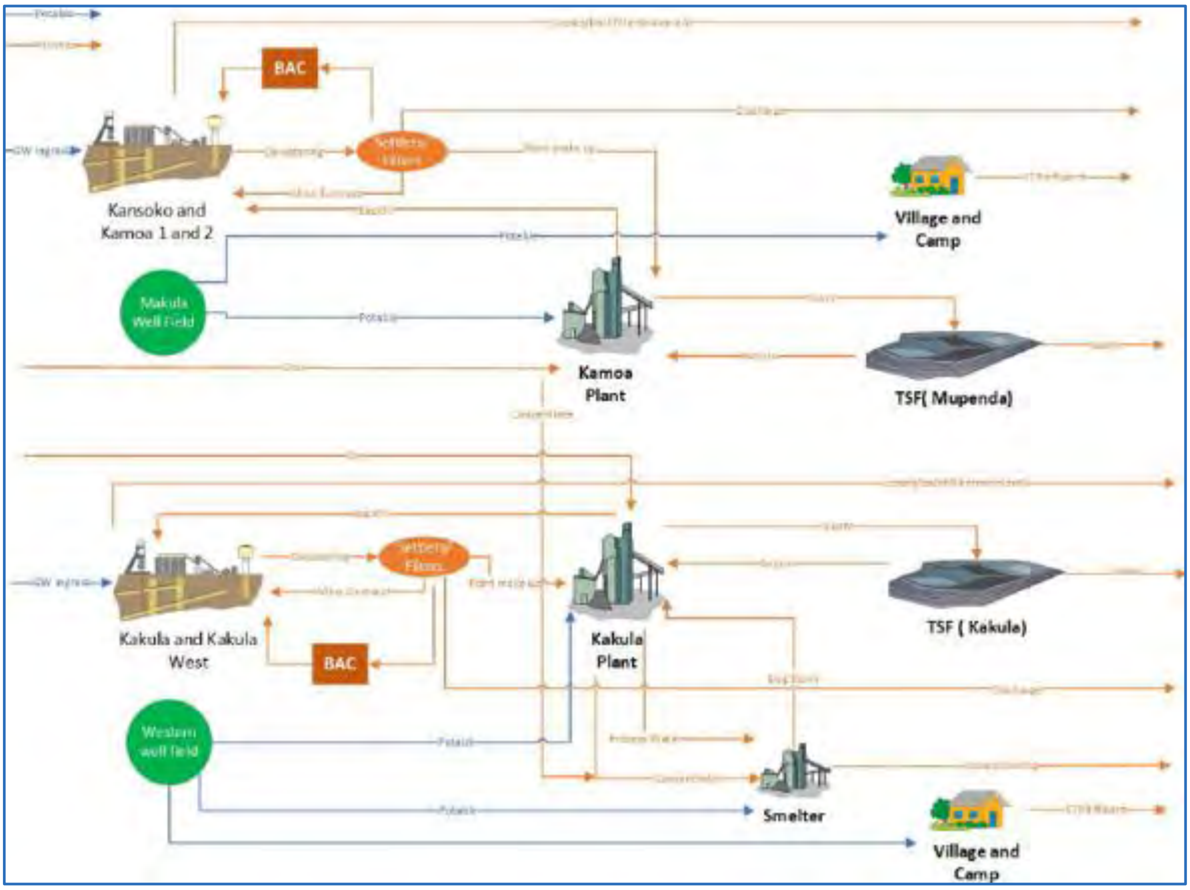
18.1.10 Water Use, Treatment and Discharge Requirements

18.1.10.1 Site Wide Water Reticulation

The PFS considers the water balance for Kamoa and Kakula areas to be made up of the same components which is the underground mining, backfill and concentrator plants, and a construction camp and village. There will be a smelter at Kakula to process the concentrate produced at the Kamoa and Kakula plants. The TSFs that will dispose of the tailings are an expansion of the current Kakula TSF. Two further cells and the Mupenda TSF will dispose of the Kamoa tailings.

The site's water reticulation is shown in Figure 18.14 and described in the sections thereafter.

Figure 18.14 Site Wide Water Reticulation



DRA, 2023.

The Kakula area consists of the current Kakula mining operation, the planned Kakula West mining area, the current Kakula Plant and planned expansion and the three cells at the Kakula TSF and the smelter. The Kamoa area consists of the Kansoko Sud, Kamoa 1 and 2 mining areas, the planned Kamoa concentrator and the proposed Mupenda TSF.

The groundwater ingress into the underground workings and the water recovered from the water sent underground for mining will be pumped to surface. The water will be settled and flocculated to achieve a water quality suitable for supplying in the mining demand and the process water make-up at the concentrators. The water pumped from underground will also be filtered for use as filtered water at the concentrators and the backfill plants. Excess water will be discharged into the river system after further treatment in environmental settlement dams to ensure that discharge meets DRC discharge standards.

The concentrators process the ore to produce a concentrate that is sent to the smelter for further processing or off-site if the smelter is not commissioned. A portion of the tailings mass is used to produce backfill which is sent underground to backfill the workings. The remaining tailings mass will be sent to the Kakula TSF and Mupenda TSF with the supernatant returned from the TSF to the concentrators for re-use.

The potable water for the Kamoa mining village, Kamoa Processing Plant and the Kansoko Sud and Kamoa 1 and 2 mining areas will be provided from the Makula Well Field. The potable water for the mining villages at Kakula, the Kakula Processing Plant, smelter, Kakula and Kakula West mining areas and smelter will be provided by the Western Well Field.

18.1.10.2 Kamoa-Kakula Water Requirements

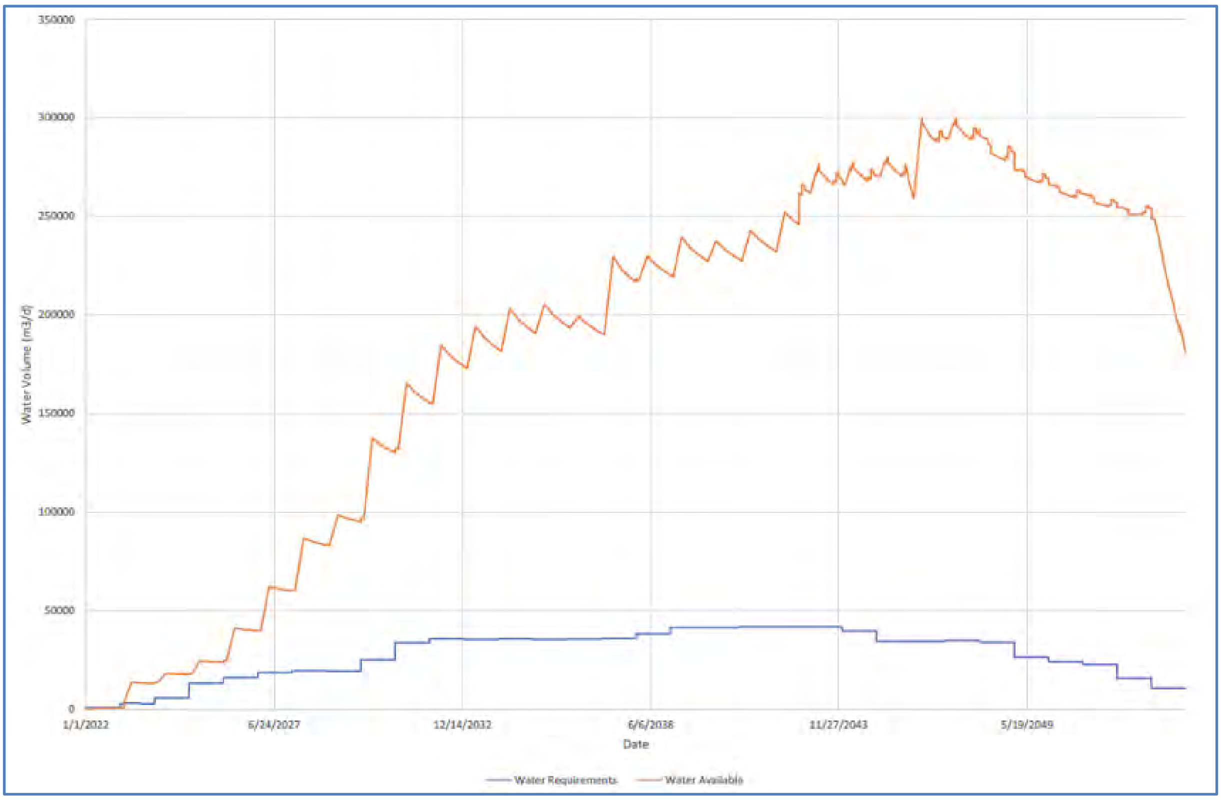
The following water requirements were identified:

- Domestic water,
- Concentrator / process water,
- Backfill water,
- Underground mining / mine service water,
- Mine dewatering,
- Ventilation water,
- Construction water,
- Ground water ingress,
- Smelter, and
- TSFs.

Kamoa Mining Area

The water requirements at the Kamoa mining area are the filter and process water required for mining, construction and at the backfill and concentrator plants. Potable water is also required and is supplied from the Makula Wellfield. The sources of water to meet the process and filtered water requirements are the water pumped from underground and return water from the TSF. The water requirements are compared to the available water in Figure 18.15. The plot shows that there is sufficient water available to supply the process and filter water requirements.

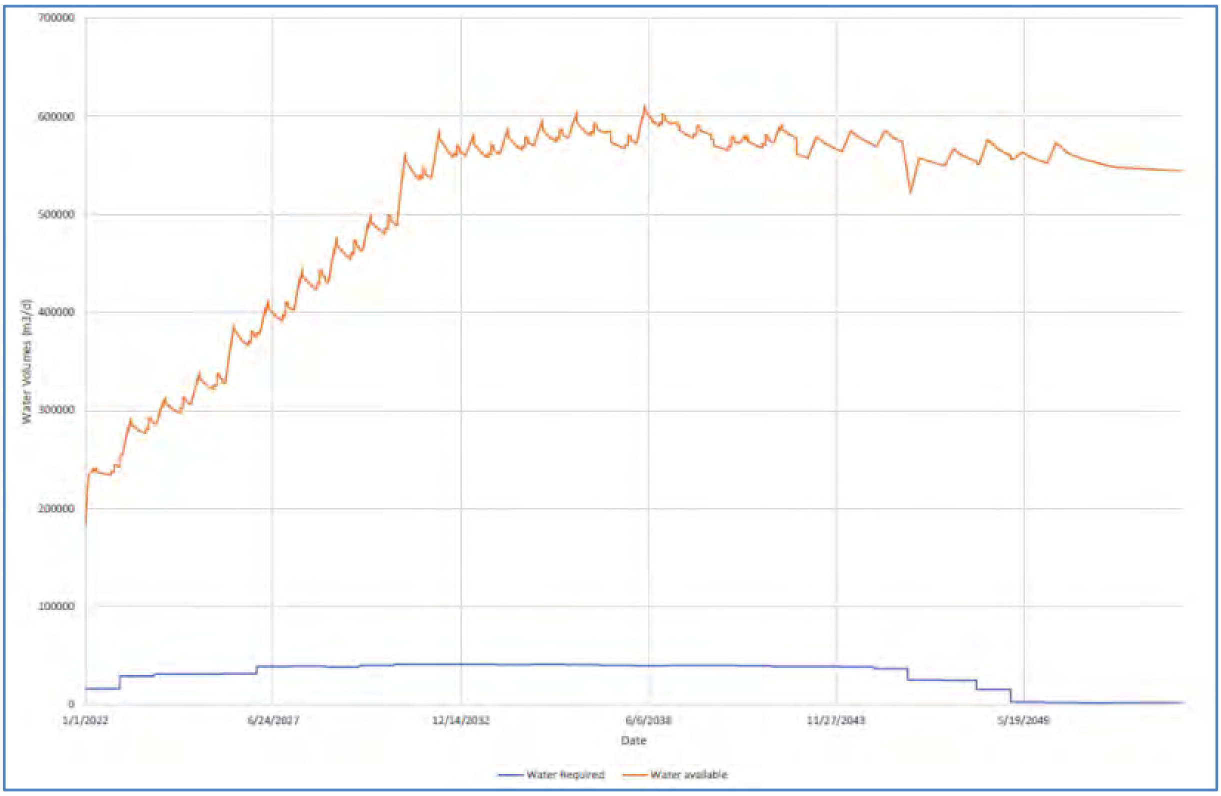
Figure 18.15 Total Process and Filter Water Required and Water Available at Kamoa Mining Area



Kakula Mining Area

The process and filter water requirements at the Kakula Mining Area include water for construction, mining, the Kakula concentrator, backfill plant and smelter. The water available to meet the filter and process water requirements are the water pumped from underground and the return water from the Kakula TSF. The water pumped from underground includes the groundwater ingress and water recovered from the water sent underground for mining. The water requirements are compared to the water available in Figure 18.16. The plot shows that there is sufficient water available to meet requirements. There are large volumes of excess water that will be discharged into the river after removing solids through the settlers on-site.

Figure 18.16 Total Process and Filter Water Required and Water Available at Kakula Mining Area



18.1.11 Water Storage Facilities

Different types of dams have been allowed for in accordance with their varied requirements. Table 18.2 indicates the water storage facilities allowed for, and their associated capacities.

Table 18.2 Water Storage Facilities and Capacities

Area	Surface / UG	Water Storage Facility	Material of Construction	Capacity	Ph3	Ph4
Kamoa						
6000	Surface	Storm Water Pond – Mining Area	Lined earth dam	15 ML	✓	
6000	Surface	Storm Water Pond – Plant Area	Lined earth dam	15 ML	✓	
6000	Surface	Construction Settling Ponds	Earth dam	8.5 ML	✓	
3000	Surface	Process Water Dam #1	Lined earth dam	15 ML	✓	
3000	Surface	Process Water Dam #2	Lined earth dam	15 ML		✓
2000	Surface	Dewatering Settling Dams x 7	Lined earth dam	60 ML x 7		✓
6000	Surface	Kansoko Borehole Water Transfer Tank	Pressed panel tank	2,000 m^3	✓	
6000	Surface	Concentrator Potable Water Tank	Pressed panel tank	1,000 m^3	✓	
6000	Surface	Surface Distribution Potable Water Tank	Pressed panel tank	300 m^3	✓	
6000	Surface	Contractors Laydown Potable Water Tank	Pressed panel tank	2,000 m^3	✓	
6000	Surface	Mining Potable Water Tank	Pressed panel tank	300 m^3	✓	
6000	Surface	Backfill Plant Potable Water Tank	Pressed panel tank	300 m^3	✓	
6000	Surface	Village Potable Water Tank	Pressed panel tank	300 m^3	✓	
6000	Surface	Makula Borehole Water Transfer Tank	Pressed panel tank	2,000 m^3	✓	
3000	Surface	Filtered Water Tank	Mild Steel	1,500 m^3	✓	
Kakula						
2000	Surface	Dewatering Settling Dams x 2	Lined earth dam	60 ML x 2		✓
Kakula West						
6000	Surface	Storm Water Pond	Lined earth dam	15 ML		✓
6000	Surface	Construction Settling Ponds	Lined earth dam	15 ML		✓
2000	Surface	Dewatering Settling Dams x 5	Lined earth dam	60 ML x 5		✓
6000	Surface	Potable Water Tank	Pressed panel tank	300 m^3		✓
6000	Surface	Fire Water Tank	Pressed panel tank	300 m^3		✓
6000	Surface	Box-cut Potable Water Tank	Pressed panel tank	300 m^3		✓
6000	Surface	Box-cut Fire Water Tank	Pressed panel tank	300 m^3		✓

18.1.12 Services

18.1.12.1 Water Treatment

The quality of excess water from the main decline water handling system will be monitored. Provision has been made for the inclusion of flocculant dosing systems to manage solids content. If necessary, it will be managed by a water treatment plant, which utilises coagulation, flocculation, lamella clarification, sand filtration and disinfection using chlorination. This process aids in the reduction of Total Suspended Solids (TSS), turbidity and organics. In addition to the excess water treatment, provision has been made for chlorination of the borehole water for drinking.

18.1.12.2 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), and
- Concrete and miscellaneous metal work (SANS 1200G).

18.1.12.3 Sewer Reticulation

The mine's domestic sewage requirements 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.

Due to the watershed, and expected sewage volumes, there will be three sewer treatment plants at Kamoa. These will be located south of the concentrator plant, the catchment of the contractor's camp, and north of the Kamoa footprint, for the mining and general admin infrastructure. One sewer treatment plant was allowed for Kakula West.

The intended sewer treatment plant is a bio sewage system, consisting of a combination of anaerobic, anoxic, micro screening and aerobic reactors to achieve an effluent low in dissolved organic compounds, and total nitrogen content.

From the collection, the sewage will be pumped via sewage pump to the above ground buffer tank and then to the process plant. From there the sewage will enter the aerobic section. In the aerobic section the pre-treated sewage will go through nitrification and denitrification before being gravity-fed to the clarification section. In the clarifier the clear liquid will enter the sterilisation zone and the sludge will be sent back to the anaerobic section for further processing.

In the sterilisation section the processed water will be polished and sterilised using plasma ozone. From the sterilisation section the (clear liquid) processed water will be sent to the discharge line.

18.1.12.4 Weighbridge

At Kamoa, four weighbridges have been allowed for. Two weighbridges will be located after the main gate before the bonded yard, and the other two will be located in the plant area to measure concentrate and materials. All weighbridges are positioned in such a way that vehicles entering / leaving the area can be weighed without disrupting traffic.

All required weighbridges at Kakula are currently in place.

18.1.12.5 TSF Pipeline Servitude

A 10 m servitude will be bush cleared for the entire route of the pipelines and a laterite road will be constructed. Three HDPE tailings pipelines will be routed along the tailing pipeline servitude to the TSF to transport waste material from the process plant to the TSF. Three more HDPE lines will be installed in the same servitude for TSF return water, back to the process plant. The pipes will be placed unsupported on the ground.

18.1.13 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.

18.1.14 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.

18.1.15 Logistics

Kamoa-Kakula's copper concentrates are currently transported by truck and shipped, either in 2-tonne bags for export or in bulk for toll smelting at the nearby Lualaba Copper Smelter (LCS) located approximately 50km away near to Kolwezi. Copper products, whether bagged concentrate, or blister copper received from LCS, is exported by truck via the ports of Durban in South Africa and Dar es Salaam in Tanzania, and to a lesser extent Walvis Bay in Namibia and Beira in Mozambique.

During the mine ramp-up, particularly since the Phase 2 concentrator declared commercial production in April 2022, the significant increase in trucking demand from Kamoa-Kakula and other operators in the Copperbelt resulted in a short-term deficit in truck availability as well as delays due to border congestion, both of which resulted in increased in-land logistics costs.

Kamoa-Kakula is working alongside its offtake partners, Zijin Mining, CITIC Metal and Trafigura, as well as the government of the DRC, to undertake initiatives to optimise the transportation of Kamoa-Kakula's products. These initiatives include working with Kamoa-Kakula's offtake partners, logistics service providers, and local entrepreneurs, to increase regional trucking capacity, improve processes for clearing products for export, and the opening of new border crossings between the DRC and Zambia.

Once the on-site DBF smelter is commissioned in line with Phase 3, truck demand for hauling copper products to port is expected to peak and gradually decrease from current levels with trucks hauling 99.7% pure blister copper, instead of approximately 50% contained copper concentrate.

The Phase 4 expansion in the Kamoa-Kakula IDP23 includes the cost of a railway spur line from Kamoa-Kakula to the main railway line near Kolwezi, which will be used to transport copper products by rail, from the mine. The railway spur will connect the Kamoa-Kakula Copper Complex directly to the anticipated Lobito Corridor, which is a railway line connecting the Angolan port of Lobito, to Zambia, and the DRC. A consortium, including Trafigura Pte Ltd, of Geneva, Switzerland, was recently awarded a 30-year concession on the Angolan side. This additional export route, once fully operational, is expected to contribute to significantly reducing in-land shipping distances, and transit times, to the ocean port of Lobito, and will further reduce the carbon footprint of Kamoa-Kakula's copper production.

On 27 January 2023, the governments of Angola, DRC, and Zambia, signed the Lobito Corridor Transit Transport Facilitation Agency Agreement (LCTTFA) in the Angolan capital Luanda. The tripartite LCTTFA is intended to coordinate the joint development activities of the Corridor and provide an alternative, strategic route to export markets for both Zambia, and the DRC.

18.1.16 Buildings

Six types of buildings will be used at Kakula:

- Containerised: Limited use of containerised buildings for early works applications only. These buildings are shipping containers, converted to a building for a specific use.
- Pre-fabricated: Limited use of pre-fabricated buildings for early works applications only. 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.
- Light steel frame system: This building method involves sections of structural wall panels made from 0,8 mm to 0,12 mm gauge, high-strength galvanised steel. These are put together using rivets or self-tapping screws to form the structural wall and roof panels, which are erected on slabs. Walls are cladded externally with fibre-cement boards and internally with fibre cement or plasterboards.
- Brick Buildings: Conventional brick buildings, constructed with blockwork brick produced locally.
- Moladi system: This building system provides a cost-effective alternative. It involves a lightweight plastic formwork system, injected with a lightweight aerated mortar mix which produces a cast in situ, steel-reinforced monolithic structure.
- 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.

Table 18.3 lists all the buildings included in the Kamoa-Kakula PFS 2023, Table 18.4 and Table 18.5 lists all the modular buildings and steel structures with civil basis and sheeting.

Table 18.3 List of Buildings

Area	Description	Type of Building	Floor Area (m^2)	Phase 3	Phase 4
Kamoa					
6000	Mine Portal Access Control Room	Containerised	95	✓	
6000	Mine Development Office	Prefabricated	668	✓	
6000	Mine Offices	Brick Building	410	✓	✓
6000	Mine Lamp Room	Light steel frame	2,130	✓	
6000	Mine Gate House - Store Area	Brick Building	40	✓	✓
6000	Mine Gate House - Red Zone	Brick Building	40	✓	
6000	Mine Change House	Brick Building	2500		✓
2000	Backfill Plant Ablution Facility	Brick Building	62	✓	
2000	Backfill Plant Gatehouse	Brick Building	40	✓	
6000	Construction Offices	Prefabricated	670	✓	
6000	Construction Laboratory	Containerised	434	✓	
3000	Plant Control Room	Brick Building	636	✓	
6000	Plant Entrance Gatehouse	Brick Building	40	✓	✓
6000	Plant Offices	Brick Building	892	✓	
6000	Plant Change House	Brick Building	1,000		✓
6000	Plant Weighbridge Office	Brick Building	52	✓	
6000	Admin Office	Brick Building	1,450	✓	
6000	Executive Offices	Brick Building	750		✓
6000	Gate House - Main	Brick Building	300	✓	
6000	Bonded Yard Building	Brick Building	442	✓	
6000	Central Receiving Offices	Brick Building	250		✓
6000	Data Centre	Brick Building	385	✓	
2000	Backfill Plant Laboratory	Brick Building	178	✓	
3000	Plant Laboratory	Brick Building	586	✓	
6000	Kitchen	Light steel frame	800		✓
6000	Dining Hall	Light steel frame	1,300		✓
6000	Kansoko Training Centre	Brick Building	4,700		✓
Kakula					
6000	Change House	Light steel frame	4,300		✓
6000	Surface Fleet Offices	Brick Building	400		✓
Kakula West					
6000	Mine Development Offices	Prefabricated	668	✓	✓
6000	Mine Offices	Brick Building	410	✓	✓
6000	Mine Lamp Room	Light steel frame	2,130	✓	✓
6000	Mine Portal Access Control Room	Containerised	95		✓
6000	Mine Gate House - Red Zone	Brick Building	40		✓
6000	Mine Gate House	Brick Building	40	✓	✓

Table 18.4 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 18.5 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	×	✓

18.1.17 Accommodation

18.1.17.1 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 has facilities, and associated infrastructure, to house 1,600 people. During Phase 3 this village will be increased in size to accommodate an additional 384 beds. This upgrade includes executive, VIP, and junior accommodation. It also includes the required ablutions, and laundry areas.

18.1.17.2 Smelter Village

As part of Phase 3, a separate village for the smelter will be built to accommodate a total of 1,600 people. The costs of this project will cover the construction of facilities such as laundry facilities, restrooms, areas for barbecuing, and housing units for junior staff, senior staff, management, and executives. Additionally, the costs will include the construction of a new kitchen and gym. Earthworks, civil engineering, road construction, and fencing are also included in the project budget.

18.1.17.3 Kamoa Village

The Kamoa Village project, as part of Phase 3, will involve the construction of around 1,600 beds, complete with necessary facilities such as kitchens, restrooms, and laundry facilities. During the Phase 4 expansion, an additional 1,600 beds will be built, utilising some of the infrastructure already in place from Phase 3.

18.1.18 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 18.6.

Table 18.6 Security and Access Control Infrastructure – Overview per typical area

	Camps and villages	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	×	✓	✓	✓	×

18.1.19 Consumables and Services

18.1.19.1 Fuel

Transport fuel, and fuelling infrastructure, is available along all 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.

18.1.19.2 Maintenance

Workshops facilities will be constructed at Kamoa, Kakula, and Kakula West 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.

18.1.19.3 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 was established. Central warehousing facilities are used, 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.

18.1.20 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).

Firewater storage will be a dedicated water supply volume, sized in accordance with the requirements of the applicable fire standard. The firewater 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 firewater pumps.

The water supply will be sized to provide the required maximum firewater flows for any single fire event. Firewater will be distributed around the plant via a firewater 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. Handheld 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.

18.1.21 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 Comments on Section 18

18.2.1 Bulk power

The assumption that the refurbishment of one of the INGA power plant turbines will be necessary to meet the project's power requirements carries some inherent risk. One of the main risks is the unavailability of the turbine for refurbishment, which could result in delays in the ramp-up of production. The requirement for additional power is expected to become necessary from 2027, and it is crucial for the project to have a reliable source of power to meet its demand.

18.2.2 Kakula Infrastructure

Infrastructure planning was completed at an appropriate level of accuracy for the Kamoa-Kakula 2023 PFS, and no issues were identified that will have a material negative impact upon the financial viability of the project.

18.2.3 Kamoa Infrastructure

Infrastructure planning was completed at an appropriate level of accuracy for the Kamoa-Kakula 2023 PFS, and no issues were identified that will have a material negative impact upon the financial viability of the project.

18.2.4 Tailings Storage Facility

Tailings produced by Kamoa will be sent to the Kakula TSF starting in 2024. Once the Kakula TSF cells reach capacity, the Mupenda TSF will be commissioned in 2041.

19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies and Offtake Strategy

Kamoa-Kakula produces a high-grade copper concentrate of approximately 50% Cu, with relatively low levels of deleterious elements that may be sold at internationally competitive concentrate terms.

In 2021, Kamoa Copper signed copper concentrate, and blister copper offtake, agreements on competitive arm's-length commercial terms with CITIC Metal (HK) Limited (CITIC Metal) and Gold Mountains (H.K.) International Mining Company Limited, a subsidiary of Zijin, for 50% each of the copper products from Kamoa-Kakula's Phase 1 production of copper per year. In 2022, Kamoa Copper entered into an amendment to the existing offtake agreements, which includes the additional production volumes from Phase 2. The revised offtake agreements with CITIC Metal and Gold Mountains are evergreen for 50% each of the production volumes from Phase 1 and 2, and include both copper concentrate, and blister copper resulting from processing of Kamoa-Kakula's copper concentrates at LCS.

In 2022, KCSA also entered into a third offtake agreement with Trafigura Pte. Ltd. (Trafigura) for a fixed volume of Kamoa-Kakula's concentrate production from 2022 to 2024, with such volume re-allocated on a pro-rata basis from CITIC Metal and Zijin. Trafigura is one of the largest physical commodities trading groups in the world and has significant experience in managing commodity logistics flows on the African continent.

All three offtakers are purchasing either the copper concentrate at the Kamoa-Kakula Mine, or the blister copper at the Lualaba Copper Smelter on a free-carrier basis, meaning the buyers are responsible for arranging freight and shipment to the final destination, which is reimbursed on an open-book basis.

Kamoa Copper's concentrates and blister copper are exported via the ports of Durban in South Africa and Dar es Salaam in Tanzania, and to a lesser extent Walvis Bay in Namibia and Beira in Mozambique.

Once the on-site DBF smelter is commissioned, IDP23 assumes that approximately 80% of the copper concentrate produced at Kamoa-Kakula will be processed at the smelter and sold as copper blister anodes of approximately 99.7% copper. The remainder will primarily be toll processed at the nearby Lualaba Copper Smelter (LCS) where a ten-year agreement is in place for approximately 150,000 wmt of concentrate, and subsequently sold as copper blister, with any outstanding concentrate sold and exported as copper concentrate.

A marketing strategy is in place for the sale of the blister anodes once the smelter is in production.

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 there will be significant new economy demand from electric vehicles, wind and solar power, as well as grid-scale improvements in transmissions and energy storage required for the changing landscape.

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 2023, 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, 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, disruptions hit record levels in 2022 of approximately 1.5 Mt primarily due to issues experienced in South America. Chilean and Peruvian production in 2022 remained lower by 8% and 1%, respectively, compared to pre-COVID levels (2019). Lower production growth from South America in 2022 was offset by production growth in the DRC and Indonesia where production was up by 26%, and 28%, respectively. Data from the International Copper Study Group (ICSG) indicates that global copper mine production in 2022 increased by approximately 3.3%, and global refined copper production in 2022 increased by approximately 3.5%. 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. S&P Global expects the copper market deficit to increase from 315,000 t in 2022, to 569,000 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 from 2023 to 2025 followed by a widening deficit from 2026 onwards. 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 $4.00/lb Cu to provide an incentive to new supply.

20 ENVIRONMENTAL AND SOCIAL

20.1 Introduction

Since the inception of the project, Kamoa Copper SA (Kamoa) has periodically undertaken the required regulatory Environmental and Social Impact Assessments (ESIA), including the related specialist studies, as well as conducted annual monitoring surveys, such as radioactivity, biodiversity, air quality, and closure liability assessments. In 2022, Kamoa appointed Congo Environment and Mining Consulting (CEMIC) Sarl to update the Environmental and Social Impact Assessment (ESIA) and Environmental and Social Management Plan (ESMP) for the Phase 3 expansion project, and to submit this to the Department for the Protection of the Mining Environment (DPEM) for approval in line with DRC regulatory requirements. CEMIC relied on previous studies undertaken by WSP Golder, formerly known as Golder Associates Africa (Pty) Ltd, (Golder), as well as data collected by themselves to inform the ESIA update. Collectively these studies and annual reports have informed Kamoa's environmental and social approach and reporting.

Ivanhoe Mines Ltd, Kamoa Copper SA (Kamoa) appointed Golder to compile the environmental and social inputs as part of the pre-feasibility study and preliminary economic assessment for the Kamoa-Kakula Mine Licence area. Golder has previously contributed environmental and social inputs to the pre-feasibility study for the Kakula Mine in 2018.

Specifically, the environmental and social work undertaken by Golder is reported in:

- Golder, 2022. Kamoa-Kakula Project: Phase 3, Submission for the NI43-101 Report - Environmental and Social. Golder Associates Africa (Pty) Ltd. Report 09 October 2022.
- Golder, 2022. Environmental and Social Impact of the Kamoa/Kakula Expansion Project: Surface Water. Report issued January 2022.
- Golder, 2022. Kamoa Groundwater Specialist Report for the EIA. Report issued January 2022.
- 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 Other 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 2019 has resulted in an updated environmental and social impact assessment (ESIA), authored by CEMIC (2022), and approved on 13 July 2022.

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 is 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 in 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 public consultations were carried out as part of ESIA updates in April 2013, October 2016, May 2019, and January 2022.

The Kamoa social and community support programme includes an ongoing consultation process with the people who are affected by the project. Kamoa is responsive to the needs of the community.

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 Alternative Plus in 2021, 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 programme 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. An eco-livelihoods programme stemming from vegetable production, to maize, and aquaculture among many others, is in place and slowly taking fundamental roots.

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$8.6M over the five-years starting in 2021 which is the effective commercial production.

Kamoa has launched several projects, in the following sectors:

- Social projects for local communities: education and adult literacy, health, and enterprise development initiatives (such as a sewing training, and brick-making programme).
- Community infrastructure: schools, community water wells, donations to various health centres, construction of houses, churches and health facilities, as part of relocation, and construction of or financing access roads.
- Community development projects: assistance with initiatives identified in Lufupa and Luilu local development programmes, entrepreneurship projects, and Kamoa's Sustainable Livelihoods Project.

The Kamoa Sustainable Eco-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 implemented a custom designed software (Isometrix software) that assists in the management of environment and social performance data and facilitates 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 by Kelkam 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. Additional assessments were undertaken by Kelkam in May 2021, and in December 2022. Furthermore, an annual radiation monitoring programme has been implemented in line with the 2018 mining regulations, and the law 017/2002 of 16 October 2002, for the protection against radiation emissions. Current mining activities pose no enhanced background radiological risk to the public and the environment.

20.6.2 Land Utilisation

A detailed soil, land use, and land capability study was undertaken by Golder in 2012–2013. Additional surveys were carried out by CEMIC as part of the 2019 and 2022 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 natural fertility of soils in the project area is marginal with most areas having poor to limited agriculture potential.

20.6.3 Water and Air Monitoring

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 discharge point in July 2019. The monitoring of the mine water discharges indicates that TSS remains the key parameter to adequately manage for mine water to be discharged into the natural environment.

Four main drainage catchments have been identified within the concession area and were included as part of the baseline surface water assessment: Tchimbundji River, Lulua River, Mukanga River, and Kamoa River. Rivers are largely driven by surface water contributions.

A baseline surface water monitoring 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. Ongoing monitoring for surface water flow and water quality is undertaken within the identified catchments, and a site wide water management strategy is being compiled.

A stormwater management plan has been developed and will be implanted along with infrastructure at all new mining sites in order to separate clean, and dirty water run-off, and to ensure dirty water is contained and managed appropriately before being released back into the environment.

Monitoring of groundwater levels commenced in 2018. Generally, groundwater within the Kamoa-Kakula mining concession area is of good quality characterised by low concentrations of dissolved ions 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 WHO 2011 drinking water quality limits. The communities located within the mining concession area rely on groundwater through a combination of shallow hand dug wells, community water supply boreholes recently drilled by Kamoa, springs and streams for their water supply needs.

Several water management strategies are in place to minimise project impacts to both surface and groundwater resources.

Air quality monitoring records for the Kamoa-Kakula project extend back to approximately 2013. A number of dust fallout and air pollutant surveys have been undertaken by Golder in 2018 and 2019, Compass Green Worldwide in 2018, and CEMIC in 2019 and 2022. Furthermore, Kamoa has implemented an annual air quality monitoring programme in line with the DRC mining regulations.

Most air pollutants fall within, or well below the standards, except for ad hoc exceedances for dust fallout. Fine dust (PM10 and PM2.5) particulates show exceedances relatively frequently and predominantly during the dry season. Particulate concentrations in any area are generally higher during the dry season, due to less natural mitigation from precipitation on wind dependent sources such as unpaved roads and open and exposed areas, but also due to an increased prevalence of wildfires and increased used of fuel for domestic heating during the winter.

A cumulative impact assessment which included the Kamoa Copper Smelter together with all other Kamoa-Kakula Project mining sources was undertaken (AQIA for the Kamoa Copper Smelter, Airshed, July 2022). Predicted concentrations of all pollutants were well below the DRC standards at all sensitive receptor locations and away from the emission sources.

20.6.4 Biological Environment

20.6.4.1 Flora Surveys

The characterisation of the terrestrial ecology is based on seven field surveys – three surveys undertaken by Golder between November 2011, and July 2013, a survey of the Kakula area by Golder in 2016, a survey undertaken by Golder in 2021 as part of the Biodiversity baseline update, and two surveys carried out in June 2019, and November 2021 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 (Dilungus)

Vegetation of the Kamoa LSA is dominated by cultivated/secondary shrubland, followed by secondary woodland. Large scale clearing of woodland by local communities for agriculture and charcoal production has occurred and continues to occur due to increased demand for fuelwood at a regional scale, and lands for subsistence agriculture at a local scale. 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. Over 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 flora species defined as range-restricted have been confirmed in the licence area during baseline studies conducted to date. No Protected Plant Species listed in Article 6 of Annex XI of the DRC 2018 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. During the 2021 survey, an orchid species that had not been previously recorded in the DRC, Habenaria hebes, was recorded growing on a dilungu in the central western extent of the licence area.

20.6.5 Fauna Surveys

The faunal assemblages of the region are generally considered depauperate, due to historical reductions of populations due to hunting pressure, as well as habitat transformation. It is therefore highly unlikely to find many of the larger mammalian taxa. A single species of conservation concern has been confirmed at Kamoa (Golder, 2014); Aardvark (Orycterapus afer), which is listed as Protected in the under Article 5 of Annexure XII of the DRC environmental legislation concerning Sensitive Habitats. 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 Broad-leaved (miombo) woodland, in Forest / Thicket, and in Grassland / Wetland. 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 diverse community with at least 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. The community composition was considered to be made up of tolerant and moderately tolerant taxa, with a few sensitive species being recorded. 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

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 Natural, Modified and Critical Habitat Assessment of the Kamoa Mine Lease Area has been undertaken by Golder in May 2021. A number of species with potential to trigger CH under criteria 1–3 were identified. However, bearing in mind the quantitative thresholds for these species, it is considered unlikely that most would trigger critical habitat designations, given their generally understood wide range and distribution. The only species confirmed to trigger critical habitat is the orchid species Habenaria hebes and Habenaria geerinckiana – since this species is found only on dilungu, this ecosystem of conservation concern is defined as Critical Habitat.

Approximately 3,183 ha of dilungu, or 92% of the original extent, remain within the local study area - there is therefore ample opportunity to protect these habitats. Further botanical survey work in the Dilungus is undertaken in order to fully characterise the different vegetation communities amongst the different dilungu systems.

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 approximately 31 ML/d for the 14.0 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. Additional boreholes are supplying water at Kansoko and Kamoa 1 mine.

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 generates large amounts of general waste as well as industrial wastes. These will be sorted to facilitate reuse and recycling. All industrial or hazardous waste are stored in secure areas. Medical and clinic waste are incinerated. A landfill site facility is 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 2010–2022.

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 43 villages or communities fall within the mine concession area, with a population estimated at least 15,000 people, and 22 of those villages would potentially be affected by the mining operations.

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. The HR and Sustainability department are revising the local community recruitment call-system to improve the uptake and keep hosting communities on board of long-term employment opportunities.

Artisanal mining also remains a subsistence activity in the Kolwezi region. However, there is no presence of artisanal miners in the immediate neighbouring of Kamoa project footprint.

The 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 programmes have shown significant household incomes for lead farmers, and local entrepreneurs involved in vegetable, maize, fishing-farming, and brick production. Kamoa is also looking at expanding these initiatives to more community members given the high demand which is 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, and a health impact assessment in 2022. The potential risks to community health, safety and security relate 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 (low-residual risk), exposure to diseases 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 quarterly basis by Kamoa Copper SA since 2020.

A consolidated noise management plan, and monitoring programme, has been developed for the wider mining rights area.

The results show that the daytime, and night-time ambient (background) levels in local communities are significantly elevated, and already typically exceed the DRC limits most of the time. Noise levels measured at the industrial monitoring locations were all below the IFC industrial daytime and night-time guideline of 70 dB(A).

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.

An additional survey was undertaken by Blast Management, and Consulting, in September 2021, concentrating on the underground operations of the Kansoko mine. Ground vibration was monitored on surface at multiple positions for the period during blasting operations at the mine. Ground vibration levels observed were less than 9 mm/s with highest frequency of levels less than 4 mm/s. These are less than general standard for safe blasting effect on structures. Ground vibrations at houses in two nearby villages, Israel and Kaponda, were also measured at 1.6 mm/s. These levels may be perceived as perceptible but are less than amplitudes required to induce damage to structures.

It is believed that the data recorded are a good baseline of typical ground vibration levels from the underground blasting operations. As the development of the mine increase in depth and the effect of blasting will reduce as depth increased.

20.6.6.3 Physical and Economic Displacement

Kamoa has developed a 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 (Kakula 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. Kakula Phase 1 affected 45 households in 15 hamlets, relocated to Muvunda village in 2018, 115 fields, and 678 fruit trees.

The Phase 2 Resettlement process was effected by Kamoa, together with the local non-governmental organisation, Alternatives Plus in July 2019, and concerned local communities affected by the construction of vent shafts. Kakula Phase 2 consists of 64 households in seven hamlets, 955 fields, and 13,098 fruit trees.

Resettlement was also required for the Kamoa to Kolwezi Airport Road, which is a provincial responsibility. Resettlement for the Kansoko, to Kakula Bypass Israel Road, affected 12 households, 10 fields, and 31 fruit trees.

The Kakula Phase 3A resettlement process was also overseen by Alternatives Plus in November 2019, and concerned local communities that were 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 Kakula Phase 3B resettlement plan, also overseen by Alternatives Plus was undertaken in February 2020 in respect of local communities that were affected by the construction of the tailings storage facility. This phase affected five 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, 2019, and 2021, as part of the original ESIA 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.

A cultural heritage management plan has been compiled and outlines the procedures to be implemented and those responsible for their implementation, in order to ensure Project impacts are mitigated and cultural resources preserved in line with best practice and national legislation. In addition, a Chance Find Procedure will allow for the recovery of cultural heritage artefacts and archaeological sites uncovered during all project phases and provide for their recording and curation.

20.6.6.5 Mine Closure Plan

A conceptual mine closure plan was developed in 2020 for the Kamoa-Kakula operations at that point in time. This plan will be updated to include subsequent phases of the life-of-mine.

WSP was appointed to update the costing model and report required for the closure costing of the current Kamoa operations, based on the immediate and life-of-mine closure scenarios for closure (as of December 2022).

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.

The Kamoa-Kakula development has caused several impacts associated with land clearance, some disruption to surface water run-off 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 Kamoa-Kakula Mine include:

- Topographic changes such as for waste rock stockpiles / TSF, and potential subsidence.

- Increased dust and noise levels.
- Loss of topsoil, soil erosion and associated sedimentation.
- Impacts on surface water due to site clearance, placement of 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 potential loss of critical habitat (watershed grasslands (Dilungus). Furthermore, given the important role of the Dilungus as sources of baseflow for streams, disruption of the Dilungus may potentially lead to impacts to surface water and groundwater, as well as surrounding ecosystems (to be assessed).

Additional key impacts on the biological environment by the Kamoa-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 Kamoa-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.
- Potential relocation of cemeteries due to project development.

20.7.4 Greenhouse Gas Intensity Metric and Benchmarking

In May 2020, a Greenhouse gas (GHG) emissions assessment was undertaken by Golder, covering scope 1 and 2 (construction and operation phases for Phase 1 of the Kamoa-Kakula project); as well as the freight of goods to the site during construction and product export as part of Scope 3 emissions. Using the basis design of the Kakula Concentrator, it was calculated that on average 393 kt Cu will be recovered annually from the Kamoa-Kakula project at full production (i.e. 12 Mtpa). This estimate was based on an average feed grade of 3.81%, and copper recovery rate of 86% (OreWin, 2019). Assuming that the Kamoa-Kakula project will emit on average 147,069 tCO_2e per annum during the operational phase, the average emissions intensity per product unit is estimated to be 0.37 tCO_2e/tCu.

A study of 19 main copper producing countries found that the average emissions primary production of copper ranged considerably from very little to 7.3 tCO_2e/tCu, with most of the production in the range of 2.5 to 3.5 tCO_2 e/tCu (Bosch and Keunen, 2009). With an average emissions intensity of 0.37 tCO_2 e/tCu, the Kamoa-Kakula project is on the lower end of the scale, which is expected given the high feed grade, and less polluting power generation (almost all of the DRC's electricity is generated from hydropower). The significance of this impact is therefore rated as negligible.

In September 2020, Kamoa also commissioned Hatch Africa (Pty) Ltd (Hatch) to conduct a GHG inventory and intensity metric estimate calculation for copper to be produced at the Phase 1 Kakula mine and surface processing complex at its Kamoa-Kakula Project. Relying on the 2020 GHG assessment undertaken by Golder, this limited study covered Scope 1 and 2 GHG emissions, and included emissions from mining vehicles, and surface vehicles, diesel consumption, emergency generator power and diesel consumption, explosives consumption as well as purchased electricity.

GHG emissions estimates were calculated on an annual basis for each year from 2023–2038 (corresponding to the mine's anticipated full capacity production). GHG emissions from the construction and closure phases of the Project were not included in this assessment, nor were scope 3 GHG data.

The average GHG intensity metric over this period was estimated to be 0.16 tCO_2e/t Cu at the mine - based on estimated total annual average emissions of 41,382 tCO_2e/yr and an average annual copper in concentrate production of 271,288 t Cu/yr.

An update of the GHG assessment to cover Phase 3 of the Kamoa-Kakula project, and additional scope 3 emissions is underway.

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), 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 and 2022).

20.7.6 Management Actions and Monitoring Programme

A management actions and monitoring programme has been compiled as part of the ESIA updates. (Golder, 2017 and CEMIC, 2019, and 2022). 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 Kamoa-Kakula 2023 PFS have been estimated for each of the following areas:

- Additional drilling.
- Underground mining.
- Additional power.
- Temporary facilities.
- Infrastructure.
- Concentrator.
- Smelter
- Indirect Costs.
- General and Administration.
- Rail.
- Transport.
- Closure.

All costs are in Q1'23 US$. Table 21.1 indicates the foreign exchange rates used in the estimate.

Table 21.1 Foreign Exchange Rate

Currencies	Rates ($)
ZAR/USD	16.00
CAD/USD	1.25
EUR/USD	0.90
CNY/USD	6.7
AUD/USD	1.35
GBP/USD	0.80

21.2 Kamoa-Kakula 2023 PFS

The Kamoa-Kakula 2023 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 21.2 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 Kamoa-Kakula 2023 PFS Unit Operating Costs

	US$/lb Payable Cu			
	2023-2024	2025-2029	First 10-Years	LOM Average
Mining	0.41	0.44	0.47	0.56
Processing	0.16	0.15	0.16	0.20
Smelter	–	0.16	0.13	0.15
Logistics	0.51	0.24	0.29	0.26
Treatment, refining and smelter charges	0.24	0.12	0.14	0.13
General and Administration	0.13	0.10	0.09	0.08
Sulfuric Acid Credits[1]	–	–0.07	–0.06	-0.07
C1 Cash Cost	1.45	1.15	1.22	1.31
Royalties and Export Tax	0.29	0.21	0.22	0.21
Total Cash Cost	1.74	1.36	1.44	1.52

Note: C1 cash costs in this table include the impact of accounting adjustments related to the addition or depletion of the surface stockpiles where applicable.
1. Acid Selling Price $150/ t Acid.

Table 21.3 Kamoa-Kakula 2023 PFS Operating Costs

	Total LOM	2023-2024	2025-2029	First 10-Years	LOM Average
	US$M	US$/t Milled			
Site Operating Costs					
UG Mining	19,380	48	40	42	41
Processing	7,167	17	15	15	15
Smelter	5,298	–	16	12	11
General and Administration	2,800	14	10	9	6
Total	34,644	79	81	78	73

Table 21.4 Kamoa-Kakula 2023 PFS Capital Cost Summary

Capital Costs (US$M)	Phase 3 Capital US$M	Phase 4 Capital US$M	Sustaining Capital US$M	Total US$M
Underground Mining				
Underground Mining	543	684	2,747	3,974
Mining Mobile Equipment	63	66	1,238	1,367
Subtotal	607	750	3,984	5,341
Power and Smelter				
Smelter Total	906	–	165	1,071
Power Infrastructure	84	134	–	218
Subtotal	990	134	165	1,289
Concentrator and Tailings				
Process Plant	262	238	193	693
Tailings	57	–	404	461
Subtotal	320	238	597	1,154
Infrastructure				
General Surface Infrastructure	662	98	150	910
Rail Spur	–	84	70	154
Subtotal	662	182	220	1,064
Indirects				
EPCM	127	141	5	273
Owners Cost	83	–	15	98
Customs Duties	92	44	175	311
Closure	–	–	145	145
Subtotal	302	185	340	826
Capital Expenditure Before Contingency	2,880	1,488	5,306	9,674
Contingency	157	65	277	499
Capital Expenditure After Contingency	3,037	1,553	5,583	10,173

Totals have been rounded.

21.2.1 Underground Mining Cost Estimates

This section describes the methods used to develop capital and operating estimates for the Kamoa-Kakula 2023 PFS. The underground costs have been compiled by DRA Projects (Pty) Ltd (DRA), Kamoa Copper and Patterson & Cooke, Golders WSP, and OreWin Pty Ltd 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
- Fixed and mobile equipment

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 and 2 execution contracts and quotations received as part of the Phase 3 basic engineering. P&G's include contractor supervision, rented equipment, consumables, accommodations, meals, transport, induction, training, wastages, etc.
- Contractors will complete all raise boring and boreholes for the life of the mine.
- 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 programme enrolled by Kamoa and the training requirements of the mining contractor.
- The phasing out of expat labour was executed in line with the ramp-up plan for each mine. During the ramp-up stage and until steady state production is reached, mostly expat labour will be used. Expat labour will be phased out two-years after steady state operations are achieved.

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)

21.2.1.3 Contingency

A 7.5% contingency was applied to the Capital estimate portion of this study. This lower than usual contingency was used due the strong basis that was used for pricing, designs and general estimating.

A 5% contingency was applied to the ventilation systems and mobile equipment fleet capital costs.

21.2.1.4 Economic Base Date

All cost estimating is in Q1'23 US Dollars (US$).

21.2.1.5 Estimate Accuracy

The capital cost estimate meets the required accuracy criteria of -15% +30% and complies with a Class 4 Estimate as defined by the Association for Advancement of Cost Engineering (AACE).

21.2.1.6 Power Cost

The power rate applied to the Kamoa-Kakula 2023 PFS operating cost model were based on a blended rate between the current actual cost of power, and what the future expansion costs would be on kWhr basis. The applied rate is 0.102 USD/kWhr.

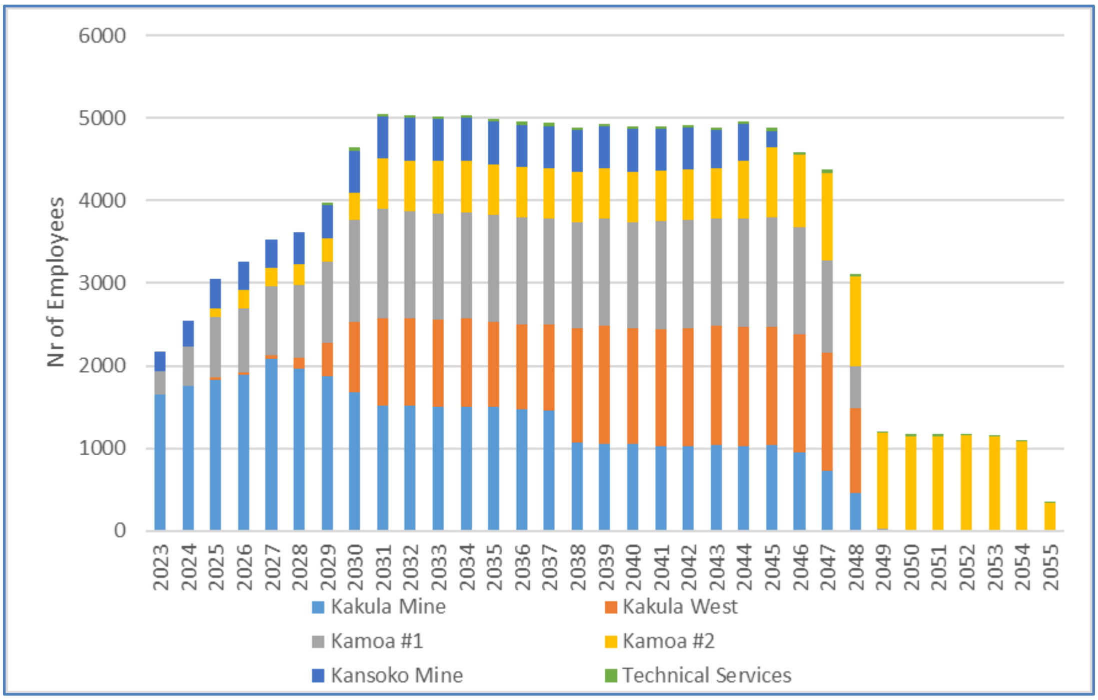
The power cost estimate was formulated using a detailed, bottom-up approach that focussed specifically on the hourly operational cost of each facility in the mine. This approach involved gathering cost data from suppliers such as consumption data on power, fuel, and lube. The hourly running cost of each facility, including tips, conveyor belts, pump stations (various), and others, was then calculated using this data. To estimate the power cost, the hourly rate for each pump, tip, conveyor, and other facility was multiplied by the relevant productivities, such as t/hr or litre/s, which were obtained from the mine production schedule and water usage.

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.

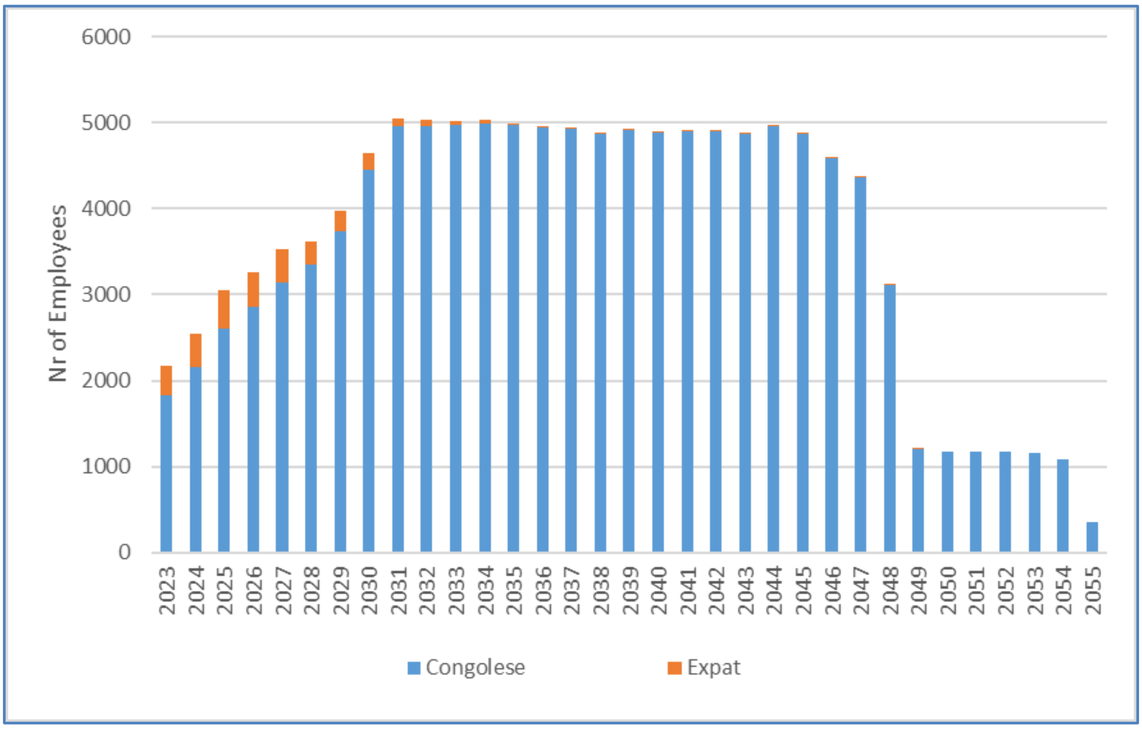
Labour estimates were derived using a bottom-up approach, aligning with Kamoa's 5-year planning for mobile equipment. This approach was applied throughout the life-of-mine for each operation. The staffing of mining infrastructure, including tips, conveyors, pump stations, and other facilities, was based on the mine schedule. The number and timing of engineering teams, supervision, and management were determined by the number of facilities, all of which were aligned with Kamoa's 5-year labour plan. The total labour compliment over LOM for Mining, and Mining infrastructure, is illustrated in Figure 21.1.

Figure 21.1 Mining Labour – Total Compliment



The labour model for the Kamoa-Kakula PFS includes a significant number of expatriate workers at the outset, with the aim of reducing operational risks and facilitating the initial setup. Over time, the expat workforce will be gradually phased out in a highly detailed manner as shown in Figure 21.2. Each expat function was evaluated to determine the most suitable phasing out plan for specific project areas. A minimum overlap period of one year, and in many cases, up to two years, was applied to ensure a smooth transfer of skills to Congolese workers. This approach ensures that the transfer of responsibilities and skills is as seamless and efficient as possible.

Figure 21.2 Mining Labour – Nationality



21.2.1.8 Mine development and stoping costs

A combination of actual costs and calculated mining costs were used to estimate mine development and stoping costs. A US$/M rate was estimated based on support requirements, excavation type (development, conveyor drive, perimeter drive, large excavation or production drift) and excavation dimensions (height and width). This linked to the production metres produced the total costs for owner mining that includes the following:

- Blasting equipment and consumables
- Drilling consumables and equipment
- Ground support - equipment and material
- Additional tools and consumables, PPE

- Services - dewatering pipes, service water pipes, EC&I, ventilation ducting
- Mucking/hauling equipment

21.2.1.9 Mobile Equipment

Mobile equipment, associated with mine development and stoping, costs include capital and operating.

Capital costs include purchase of equipment and rebuilds. Mobile equipment quantities and operating hours were calculated based on 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 rebuild and replacement frequencies over LOM.

Mobile equipment rebuilds 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:

- Only primary fleet is rebuilt and rebuild life is five-years.
- Rebuild cost is assessed at 50% of the base unit cost.
- Replacement cost is assessed at 100% of the base unit cost plus options and freight.

Operating costs includes fuel, tyres, parts and maintenance were all based vendor supplied Life Cycle Costing information and database consumption figures.

21.2.1.10 Fixed Equipment

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 were 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.

21.2.1.11 Backfill/Paste Fill

A backfill consultant provided cost estimate data for the backfill. A first principal 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.
- Paste fill.

All backfill costs included in the Kamoa-Kakula 2023 PFS 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 compiled from actual costs at Kakula for 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 Kamoa-Kakula 2023 PFS 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 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-tyred 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 tyres).

Engineering, Procurement, and Construction Management Allowance

The current Phase 3 EPCM proposal from all the various consultants were added together and used as a % for the Phase 4 expansion. The calculated % was 7% of total capital 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, travelling 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 Phase 3 and 4 construction phases. The team and costs were provided by Kamoa Copper.

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. 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%.

Afridex Blasting Costs – DRC

Afridex blasting costs are included in the Owner indirects. Zero based estimating determined the appropriate quantities. 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 was applied to account for assistants that will be employed by the mine.

21.2.2 Concentrator and Site Infrastructure Costs

The cost estimate for upgrading the Kakula plant to 9.2 Mtpa, and the related surface infrastructure for Phase 3 is based on well-defined sources. This estimate is derived from a completed Phase 3 Basic Engineering study, and the Project Forecast Cost to Completion of the ongoing construction work at the Kakula concentrator (debottlenecking), and surface infrastructure. All capital costs for the Kakula plant beyond the upgrade of the concentrator have been classified as sustaining capital.

The cost estimate for Kakula West has taken into consideration the same rates and quantities as required to bring this operation into production. The surface layout, terraces, buildings, and other infrastructure at Kakula West have been based on the construction work at Kakula and Kamoa, and the list of facilities required has taken into account that much of the Kakula infrastructure will be used for Kakula West.

21.2.2.1 General

The Kamoa-Kakula PFS capital cost estimate meets the required accuracy criteria of -15% +30% and complies with a Class 4 Estimate as defined by the Association for Advancement of Cost Engineering (AACE). The estimate has been presented in June 2022 US$.

The following inputs and documents were used in compiling the estimate that includes surface infrastructure, concentrator plant, and backfill plant:

- 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.

21.2.2.2 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. All earthworks costs were based on the updated geotechnical work, and recommendations provided by Knight Piésold.

All other surface infrastructure were costed according to the specifications provided in Section 18.

21.2.2.3 Conveyor Earthworks

No bulk earthworks and terracing are required for conveyors, only restricted earthworks for civil bases and sleepers.

21.2.2.4 Structural Steelwork and Platework Supply and Erection

For the structural, mechanical, platework, supply and erect aspect of the project, Area 2000's structural steel, platework and conveyor steel quantities have been derived from the underground general arrangement layout drawings, the 3D model, and the MEL. The BOQ for these materials was priced by Louwil and KKCC for supply and installation.

For the concentrator, the structural steel quantities were taken from the Phase 1 actuals supplied by the Project QS, and the platework quantities were remeasured. New conveyor designs were created for Phase 3 and the Project QS supplied benchmarked rates for the BOQ. The Phase 4 concentrator was a duplicate of Phase 3, with the exception of some common areas such as crushing and screening.

The Kakula backfill plant uses rates from the current contractor Modern Heavy, while the Kamoa #1 Ph1, Ph2, Kakula West Ph1 and Kakula West Ph2 backfill plant rates were aligned with the Kamoa Concentrator plant.

21.2.2.5 Mechanical Equipment

All areas used the MELs to gather information on mechanical equipment and allocate them to procurement packages. Vendors were approached for pricing and historical costs were used when necessary, but mostly backed up by vendor quotations and revalidated rates. Rates for installation were based on executed contracts, placed contracts or calculated on a rate per tonne basis and applied to the new equipment required. The Client's QS team and the execution contractor provided the installation rates.

21.2.2.6 Belt Conveyors

The designs of all belt conveyors were done by the DRA engineering department, including calculations for each system and quantification of mechanical conveying equipment components and steelwork. These were compiled into unpriced schedules and sent to selected equipment vendors as a formal Request for Quotation (RFQ). The designs were based on the belt profiles as per the general arrangement drawings and were made to meet the process requirements and general engineering design criteria.

21.2.2.7 Piping and Valves

The basis of estimate for piping and valves for the different areas was determined using a combination of 3D models, general arrangement drawings, P&IDs, and BOQs from current contractors. For Area 2000, the piping was quantified using a material take-off from the Phase 2 3D model, and rates were obtained from KKCC. The piping for the Backfill Plant was quantified using a material take-off from the 3D model, and 2D drawings and rates were also obtained from KKCC. Area 3000 used BOQs developed from the concentrator P&IDs and drawings and rates were a combination of current South African contractor's rates and recommendations from two proven suppliers, Austruc and Citic. For Area 6000, the surface, overland, process, and services piping was quantified from the Block plan and rates were obtained from South African suppliers based on BOQs. The mechanical and automated valves were quantified from the latest valve list for Phase 3 Basic Engineering and valve supply costs were sourced from budget quotations from the market.

21.2.2.8 Electrical, Control and Instrumentation

Electrical loads were assigned to all transformers and Motor Control Centre (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. MV related infrastructure are costed under A6000, and LV, MCC's are allocated to the relevant A3000 or A2000 level 6 WBS.

The following basis was used for the items specified below:

- Cables: The MEL was used to determine the size and quantities of motor control centre (LV) cables. Average length of 180 m has been allowed per drive / motor run from the MCC. Medium Voltage (MV) and overland cables were measured as per the SLD from the block plans and mining design.
- Cable Racking: Where designs are not applicable or available, cable racking quantities was estimated based on the cable lengths.
- Field Isolators: Based on the preliminary motor count. Same design parameters and supply figures were used as per historical values (Phase 1 and Phase 2) with an escalation clause.
- MCC and Starters: Quantities as per the MEL. Rates were quoted from the same supplier as was used during Phase 1 and Phase 2.
- Transformers: Quantities as per the SLDs. Rates as per the Phase 1 contract.
- MV Motors, VSDs and Phase Shift Transformers quantities as per the SLD. Cost as per RFQ pricing.
- MV Switchgear: 33/11 kV quantities as per the single line diagram. Cost as per RFQ pricing.
- 220/33 kV, 80 MVA transformer was costed based on the same specification as the Phase 1 transformers. The Contract rates for Phase 1 was used. Lighting and Small Power: no detail engineering involved, costed on a per square meter basis for each area.
- Installation cost: Rates as per the Phase 1 and Phase 2 contract were used.

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; these diagrams formed the basis for the quantities of control equipment. Costs for the C&I equipment are as per Phase contracts except where revised rates were obtained. Revised rates were received from equipment vendors for the following equipment types:

- Flow transmitters.
- Level and pressure transmitters.
- Conveyor safety instrumentation.
- Belt scales.
- Siemens control system hardware (RIO panels, SCADA server panels, HMI panels, PLC panels, ASI panels and junction boxes).

- CCTV – quantities and rates for process cameras were obtained from the Phase 1 project BOQ. The rates were escalated to account for inflation.

21.2.2.9 Standby Power

The following additional generators will be installed to provide emergency power:

- Eight additional gensets will be installed at the 33 kV Kakula KCS substation.
- Six 3.2 MW generators to be installed at the Kansoko 33 kV substation.
- Ten 3.2 MW generators to be installed at 33 kV Kamoa 1 substation.
- Four 3.2 MW generators to be installed at 33 kV Kamoa 2 substation.
- Ten 3.2 MW generators to be installed at 33 kV Kakula West substation.

The rates are based on a recent contract that was placed on these generators.

In addition to the above a 42 MW standby facility was included in the cost estimate based on a RFQ.

21.2.2.10 Transportation

A Tonnage per load basis was applied to estimate transport costs across all areas in the estimate. Certain disciplines such as buildings, civils and earthworks had supply, transport and erect costs in the rates as provided by the contractors. For other disciplines the following costs were used:

Ocean Freight (40 ft Container):

- China to Site- $25,000
- Europe to Site- $18,000
- Finland/Sweden to Site- $20,000
- USA to Site- $18,000

Road Freight (Tri Axle):

- Durban Port to Site- $ 10,050
- Johannesburg to Site- $ 9,900

The load factors applied to each 40 ft shipping container was applied as follows:

- Steelwork (t)- 15.6
- Mechanical (t)- 14.2
- Platework (t)- 10.3
- Piping (t)- 19 – 21.2
- EC&I- 12.5% of supply cost

The transport costs were calculated and added to the associated supply costs.

21.2.2.11 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
- Wastewater treatment plant
- Laboratory equipment

21.2.2.12 Spares

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
- Mining spares were estimated on a quantity basis such as meter conveyor, number of spare pumps and drives etc.

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
- Valves – 5.0% of piping and valves supply

Operational Spares:

- Operational spares were excluded from the estimate

21.2.2.13 Construction Facilities Costs

Allowances have been made in the capital cost estimate for the following during the construction period:

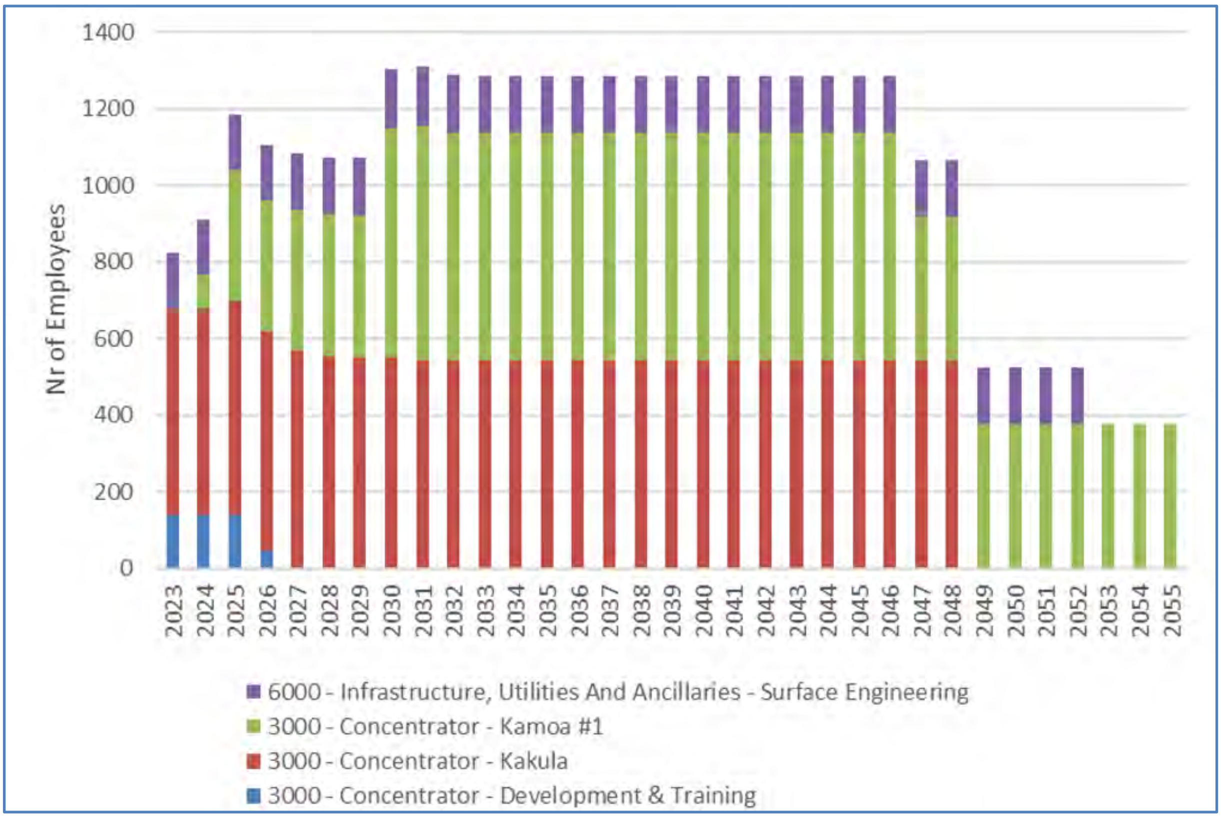
- Construction power
- Construction water
- Construction laydown area
- Construction offices
- Construction communication
- Construction vehicles
- Construction SHEQ
- Construction signage
- Construction ablution facilities
- Construction IT and computer equipment
- Construction access and security facilities
- Construction waste facilities
- Commissioning tools

21.2.2.14 Labour

The same cost basis for labour as discussed in Section 21.2.1.7 was used for the concentrator, and surface infrastructure.

The labour requirements for the concentrator were estimated on a per-module basis. Specifically, labour requirements for Kakula Phase 1 and 2 were aligned with the current labour planning and extrapolated to cover the labour needs for Kamoa modules 1 and 2. The infrastructure labour was aligned with the site's 5-year labour plan, and each function was evaluated to extrapolate labour needs over the life of the project. The total labour compliment for concentrator and infrastructure is illustrated in Figure 21.3.

Figure 21.3 Concentrator and Surface Infrastructure - Total Labour Compliment



Development and training labour were obtained from the current site onboarding plan and included in the PFS labour model.

21.2.2.15 Power

The power cost estimation for the concentrator and infrastructure was conducted based on the mechanical equipment list. The power consumption for each piece of equipment was determined, and then linked to the equipment's running time per-day to create a power consumption profile. In addition, the power draw for the mill was obtained through simulation studies.

Standby power costs are the diesel cost to run standby generators for an assumed number of hours per week. As the over network stability is expected to improve the running hours for Phase 3 was assumed to be nine hours per week, and Phase 4 was reduced to five hours per week. The diesel consumption per hour is based on the total installed generation capacity.

In addition, running cost for a new 40 MW standby generator plant was also included in the financial model. Diesel consumption was calculated based an assumed running time of three hours per day (1,000 l consumption per hour).

21.2.2.16 Consumable

Concentrator plant consumables were based on consumption specified in the process design criteria which in turn was obtained from testwork. Kakula consumption figures were aligned with site data. All costs were obtained from current supply costs on-site.

The estimated maintenance costs for the bulk power, general surface infrastructure, water treatment, and sewage plant were calculated based on a percentage of their capital supply costs. The percentage is determined by the expected useful life of the infrastructure, which varies depending on the specific discipline. For instance, buildings are assumed to have a useful life of 15-years, while electrical equipment is assumed to have a useful life of 10-years, and piping and valves are assumed to have a useful life of five-years. After the percentage is determined, it is applied to the estimated operating cost on a per annum basis average. This methodology allows for the maintenance costs to be spread out over the useful life of the infrastructure, ensuring that the costs are accurately reflected in the project budget.

21.2.3 Owner's Cost and G&A

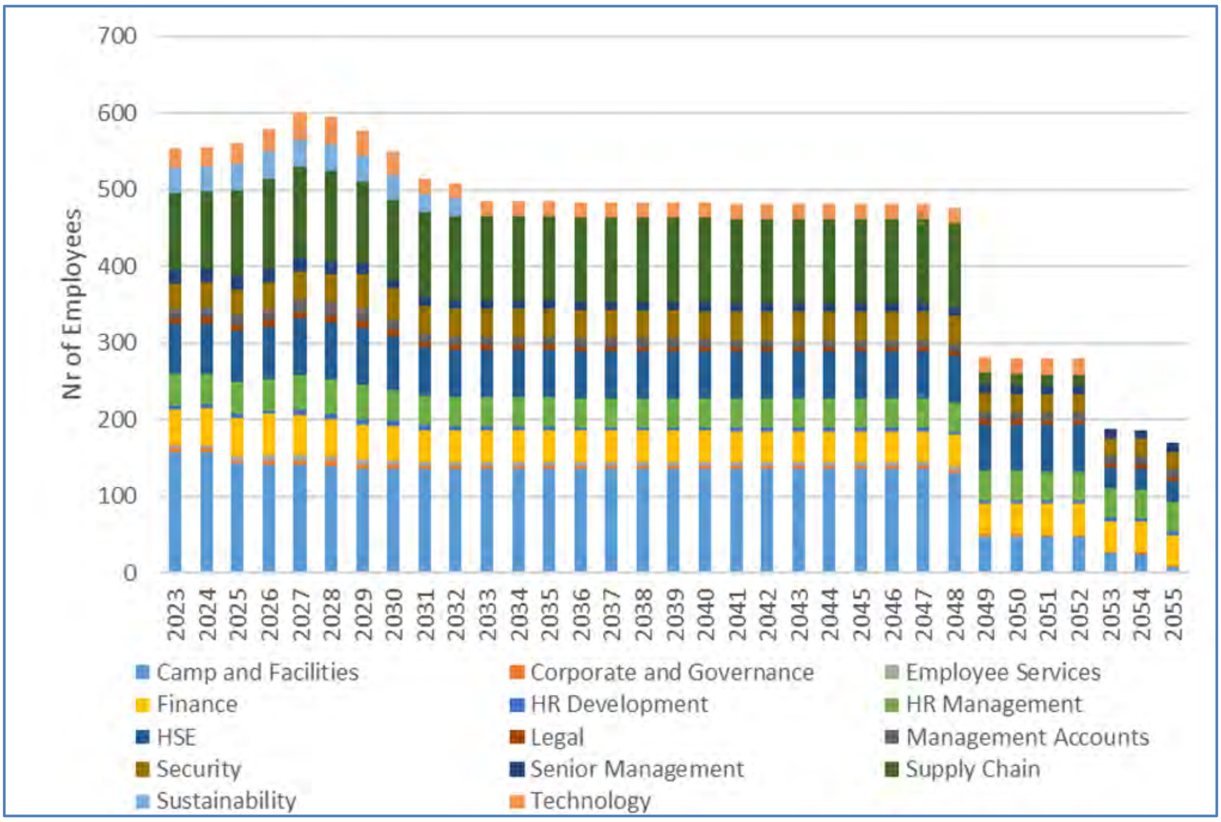
Kamoa Copper SA have prepared a budget for Owners costs these were reviewed and adjusted to allow for capital and operating costs for the life-of-mine estimate. The costs include allowance for the following items:

- 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.

The labour estimate for G&A is illustrated in Figure 21.4.

Figure 21.4 G&A Labour – Total Compliment



21.3 Comments on Section 21

In the opinion of the QPs, the work completed for the Kamoa-Kakula 2023 PFS adequately support the study. 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

22.1 Economic Assumptions

The modelling and taxation assumptions used in the Kamoa-Kakula 2023 PFS are 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 Kamoa-Kakula 2023 PFS Pricing and Rates Summary

Model Assumption	Value
Copper price (US$/lb)	3.70
Concentrate treatment charge (US$/t concentrate)	88
Blister refining charge (US$/t Blister)	175
Concentrate refining charge (US$/lb Cu)	0.088
Transport (US$/t concentrate)	395
Copper payability (%)	96.75
Provincial Export Road Tax (US$/t concentrate)	50
DRC 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"	1.0

The copper price used in the economic analysis is $3.80/lb in 2023, $3.90/lb in 2024, $4.00/lb in 2025, $4.00/lb in 2026, and a long-term copper price of $3.70/lb from 2027 onwards.

The Project level valuation model begins on 1 January 2023. It is presented in Q1'23 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.

The mineral products are assumed to be sold on a Free Carrier (FCA) basis. Therefore, Kamoa Copper would not incur transportation/freight costs, as the buyer would pay for these costs directly.

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, with the use of the carried forward losses limited to 60% of the tax result of the considered year. The aggregate exploration expenditure may be claimed.

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.

Export Tax on Concentrate

A tax on the export of concentrate is levied on a per tonne basis and equates to US$100/t NSR 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 moveable 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.

Tax on Excess Profits

A special tax on excess profits applies when prevailing commodity prices are more than 25% higher than those prices used in a feasibility study filed with the DRC tax authorities. A tax of 50% is levied on such incremental profits, from which income tax payments are deductible.

22.1.1.5 Sunk Costs

The estimate excludes all sunk costs up to 31 December 2022.

22.2 Kamoa-Kakula 2023 PFS Overview and Results

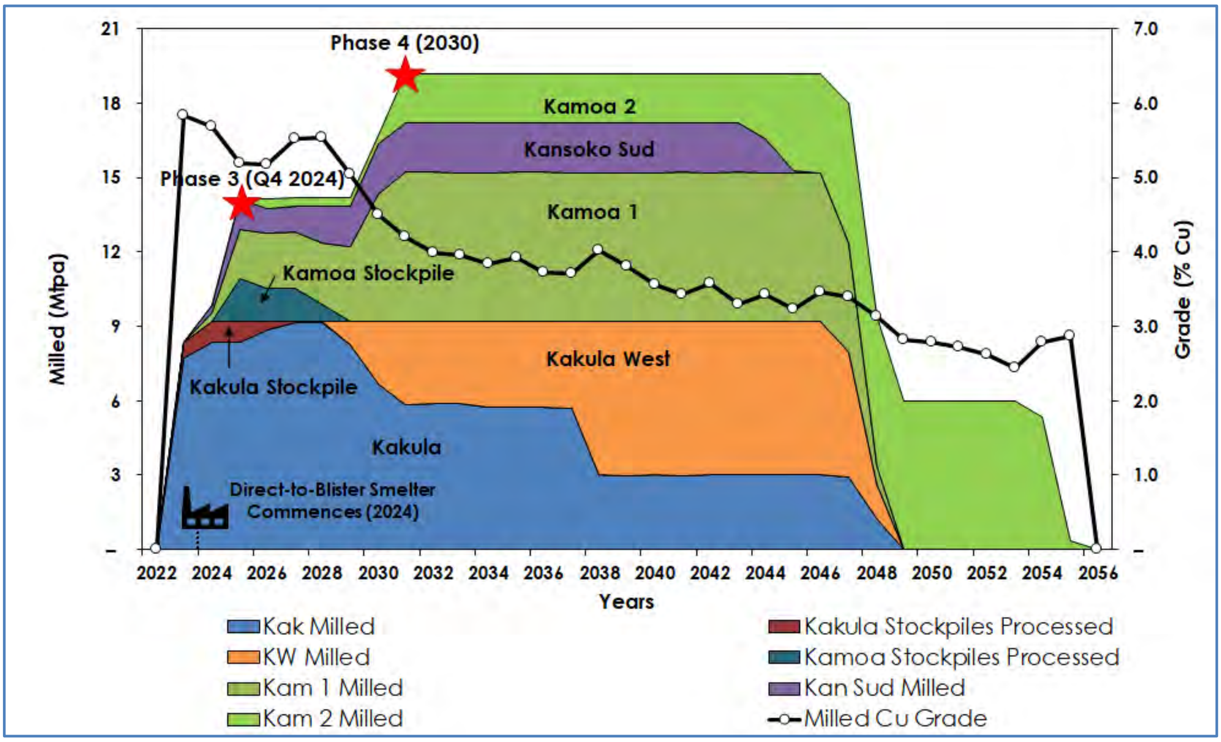
The Kamoa-Kakula 2023 PFS analyses a production case with an expansion of the Kamoa-Kakula concentrator processing facilities, and associated infrastructure to 19.2 Mtpa, and includes a smelter, and five separate underground mining operations with associated capital and operating costs. The five mines are listed below.

- Kakula Mine (PFS 9.2 Mtpa).
- Kakula West Mine (PFS 6.2 Mtpa).
- Kamoa 1 Mine (PFS 6.0 Mtpa).
- Kansoko Sud Mine (PFS 2.0 Mtpa).
- Kamoa 2 (PFS 6.0 Mtpa).

The LOM production scenario provides for 472.3 Mt to be mined at an average grade of 3.94% copper, producing 37.8 Mt of high-grade copper concentrate, containing approximately 35.1 billion pounds of copper over a mine life of 33-years.

The expansion of Kamoa-Kakula concentrator processing facilities occurs in two phases (Phase 3 and Phase 4). Phase 3 includes the ongoing construction of a new 5.0 Mtpa concentrator, located at Kamoa, planned for completion in Q4'24. As part of the Phase 3 expansion, a direct-to-blister (DBF) flash smelter is under construction to produce approximately 500,000 tonnes-per-annum of 99+% pure copper metal. Phase 4, planned for completion in 2030, includes the construction of an additional 5.0 Mtpa concentrator in parallel to Phase 3, which will be fed by mines in the Kamoa area, bringing overall production up to 19.2 Mtpa. The long-term development scenario for Kamoa-Kakula PFS 2023 is shown in Figure 22.1.

Figure 22.1 Kamoa-Kakula 2023 PFS Long-Term Development Scenario



OreWin, 2023.

22.2.1 Summary of Key Physical and Financial Metrics – Kamoa-Kakula

A summary of the key results for the Kamoa-Kakula 2023 PFS scenario are:

- The initial 10-years of production is projected to have an average grade of 4.94% copper, resulting in estimated average annual copper production of 620,000 tonnes.
- Phase 3 capital, including contingency, is estimated at US$3,037M, from 1 January 2023.
- Phase 4 capital, including contingency, is estimated at US$1,553M, from 1 January 2023.

- Average total cash cost of US$1.44/lb of copper during the first 10-years.
- After-tax NPV, at an 8% discount rate of US$19.1 billion.

The key results of the Study are summarised in Table 22.2

Table 22.2 Kamoa-Kakula 2023 PFS Results Summary

Item	Unit	Total
Total Processed (life-of-mine)		
Ore Milled	kt	476,195
Copper Feed Grade	%	3.94
Total Concentrate Produced (life-of-mine)		
Copper Concentrate Produced	kt (dry)	37,802
Copper Recovery	%	86.62
Copper Concentrate Grade	%	43.05
Contained Metal in Concentrate	Mlb	35,875
Contained Metal in Concentrate	kt	16,273
Annual Average (2023-2024)		
Ore Milled	kt	9,106
Copper Feed Grade	%	5.75
Copper Concentrate Produced	kt (dry)	917
Contained Copper in Concentrate	Mlb	1,004
Contained Copper in Concentrate	kt	455
C1 Cash Cost	US$/lb. payable	1.45
EBITDA	US$M	2,015
Annual Average (2025-2029)		
Ore Milled	kt	14,194
Copper Feed Grade	%	5.30
Copper Concentrate Produced	kt (dry)	1,431
Contained Copper in Concentrate	Mlb	1,442
Contained Copper in Concentrate	kt	654
C1 Cash Cost	US$/lb. payable	1.15
EBITDA	US$M	3,522
Annual Average (First 10-Years)		
Ore Milled	kt	14,428
Copper Feed Grade	%	4.94
Copper Concentrate Produced	kt (dry)	1,379
Contained Copper in Concentrate	Mlb	1,368
Contained Copper in Concentrate	kt	620
C1 Cash Cost	US$/lb. payable	1.22
EBITDA	US$M	3,151
Key Financial Results		
Remaining Phase 3 Capital Costs	US$M	3,037
Phase 4 Capital Costs Capital Costs	US$M	1,553
Sustaining Capital Costs	US$M	5,583
LOM Avg. C1 Cash Cost	US$/lb Payable Cu	1.31
LOM Avg. Total Cash Costs	US$/lb Payable Cu	1.52
LOM Avg. Site Operating Costs	US$/t Milled	72.75
After-Tax NPV8%	US$M	19,062
Project Life	Years	33

Table 22.3 summarises the financial results, whilst Table 22.4 summarises mine production, processing, concentrate, and metal production statistics.

Table 22.3 Kamoa-Kakula 2023 PFS Financial Results

	Discount Rate (%)	Before Taxation	After Taxation
Net Present Value (US$M)	Undiscounted	67,966	47,969
	4.0%	41,321	28,966
	6.0%	33,325	23,272
	8.0%	27,407	19,062
	10.0%	22,937	15,884
	12.0%	19,493	13,438
Project Payback Period (Years)		1.2	1.6

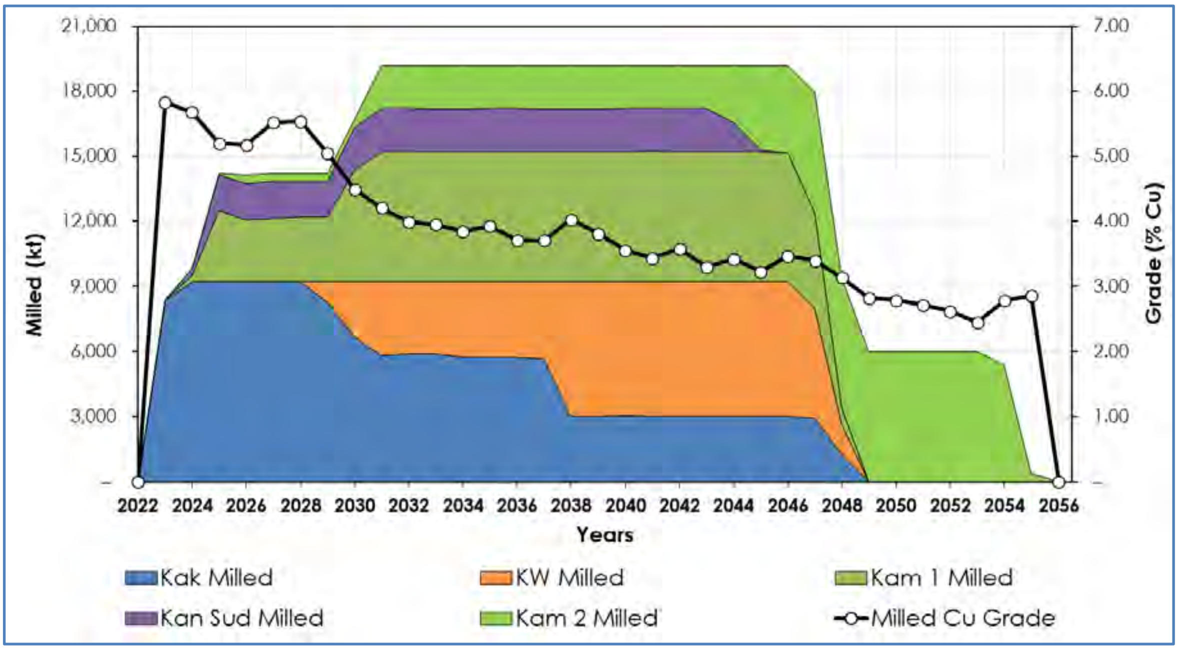
Table 22.4 Kamoa-Kakula 2023 PFS Production and Processing

Item	Unit	2023-2024	2025-2029	First 10-Years	LOM Average
Total Ore Processed					
Quantity Milled	kt	9,106	14,194	14,428	14,430
Copper Feed Grade	%	5.75	5.30	4.94	3.94
Total Concentrate Produced					
Concentrate Produced	kt (dry)	917	1,431	1,379	1,146
Recovery	%	86.97	87.02	87.02	86.62
Concentrate Grade	% Cu	49.67	45.70	45.01	43.05
Copper in Concentrate					
Contained Copper	Mlb	1,004	1,442	1,368	1,087
Contained Copper	kt	455	654	620	493
Concentrate Smelted / Sold					
Concentrate Smelted (Kamoa)	kt (dry)	–	1,133	936	861
Concentrate Tolled (LCS)	kt (dry)	134	134	134	120
Concentrate Sold	kt (dry)	783	164	310	165
Payable Copper Sold					
Blister Anodes (Kamoa)	kt	–	496	396	353
Blister Copper (LCS)	kt	64	65	63	55
Copper in Concentrate	kt	376	80	147	75
Payable Metal					
Copper	Mlb	971	1,411	1,336	1,064
Copper	kt	440	640	606	483

Note: The 2023-2024 average includes approximately 20 kt of copper in concentrate that is processed by the Phase 3 concentrator during the ramp-up period in 2024.

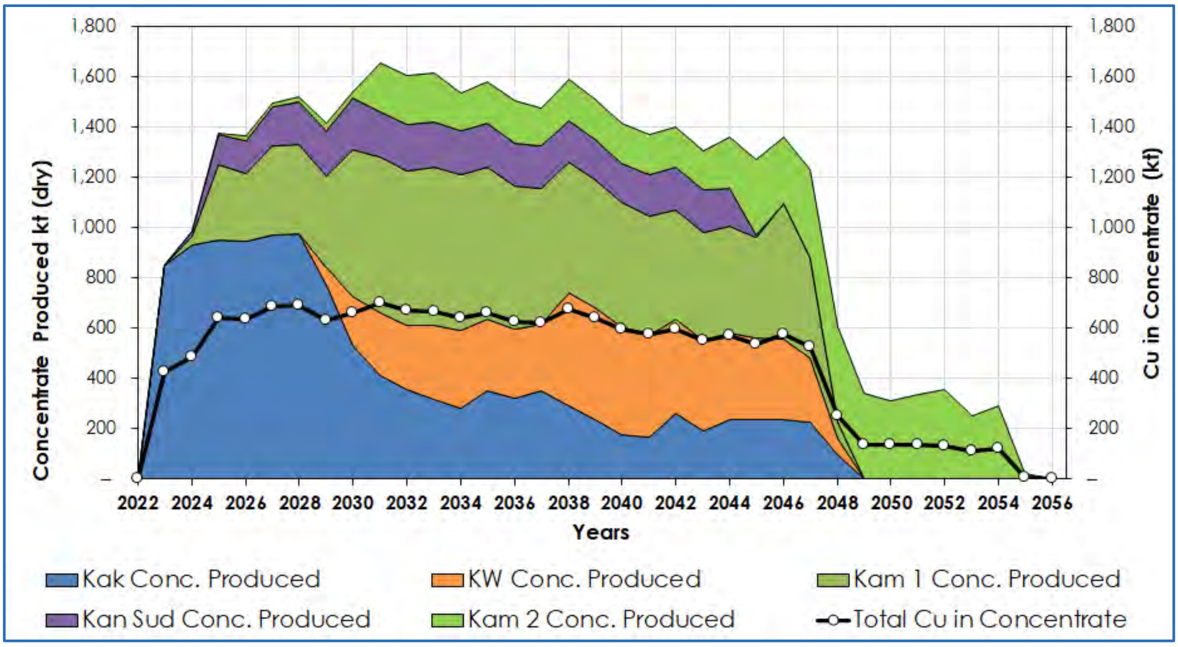
The Kamoa-Kakula 2023 PFS mill feed and copper grade profile and the concentrate and metal production for the LOM are shown in Figure 22.2 and in Figure 22.3.

Figure 22.2 Kamoa-Kakula 2023 PFS Process Production



OreWin, 2023.

Figure 22.3 Kamoa-Kakula 2023 PFS Concentrate and Metal Production



OreWin, 2023.

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 Kamoa-Kakula 2023 PFS Unit Operating Costs

	US$/lb Payable Cu			
	2023-2024	2025-2029	First 10-Years	LOM Average
Mining	0.41	0.44	0.47	0.56
Processing	0.16	0.15	0.16	0.20
Smelter	–	0.16	0.13	0.15
Logistics	0.51	0.24	0.29	0.26
Treatment, refining and smelter charges	0.24	0.12	0.14	0.13
General and Administration	0.13	0.10	0.09	0.08
Sulfuric Acid Credits[1]	–	–0.07	–0.06	–0.07
C1 Cash Cost	1.45	1.15	1.22	1.31
Royalties and Export Tax	0.29	0.21	0.22	0.21
Total Cash Cost	1.74	1.36	1.44	1.52

1 Acid Selling Price $150/ t Acid.

Table 22.6 Kamoa-Kakula 2023 PFS Revenue and Operating Costs

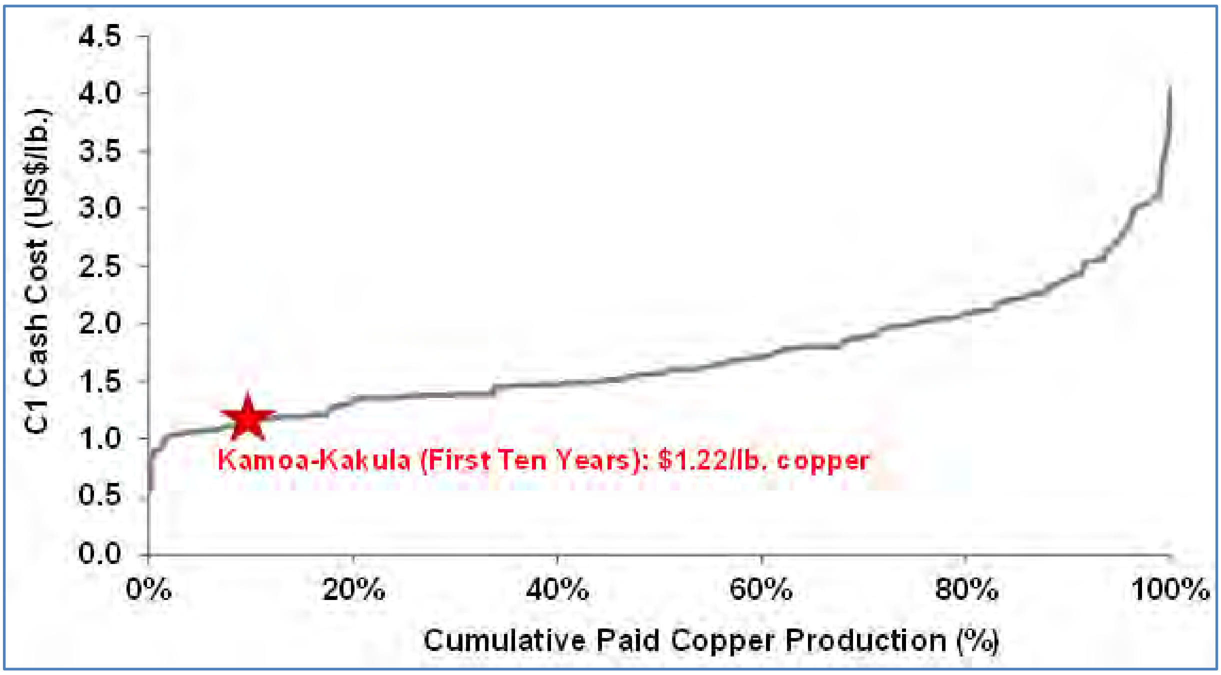
	Total LOM	2023-2024	2025-2029	First 10-Years	LOM Average
	US$M	US$/t Milled			
Revenue					
Copper in Blister	110,500	60	333	265	232
Copper in Concentrate	20,569	351	47	85	43
Acid Production	2,618	–	7	6	5
Gross Sales Revenue	133,687	411	386	356	281
Less: Realisation Costs					
Logistics	9,053	55	24	27	19
Treatment, refining and smelter charges	4,719	25	12	13	10
Royalties and Export Tax	7,417	31	21	21	16
Total Realisation Costs	21,189	111	57	60	44
Net Sales Revenue	112,498	300	329	296	236
Site Operating Costs					
UG Mining	19,380	48	40	42	41
Processing	7,167	17	15	15	15
Smelter	5,298	–	16	12	11
General and Administration	2,800	14	10	9	6
Total	34,644	79	81	78	73
EBITDA	77,854	221	248	218	163
EBITDA Margin (%)	58.2%	53.9%	64.3%	61.3%	58.2%

Note: The copper price used in the economic analysis is $3.80/lb. in 2023, $3.90/lb. in 2024, $4.00/lb. in 2025, $4.00/lb. in 2026 and a long-term copper price of $3.70/lb. from 2027 onwards.

The C1 pro rata copper cash costs of the Kamoa-Kakula 2023 PFS on Wood Mackenzie's industry cost curve is shown in Figure 22.4. C1 pro rata cash costs that reflect the direct cash costs of producing paid copper incorporating mining, processing, mine site G&A and off-site realisation costs, having made appropriate allowance for the costs associated with the co-product revenue streams.

The comparison of cash costs on the chart was not reviewed by Wood Mackenzie prior to filing, and information was sourced from public disclosures.

Figure 22.4 C1 Pro-rata Copper Cash Costs



Ivanhoe, 2023. Source: Wood Mackenzie 2023.

22.2.3 Capital Costs

The capital costs for the project are summarised in Table 22.7.

Table 22.7 Kamoa-Kakula 2023 PFS Capital Costs

Capital Costs (US$M)	Phase 3 Capital US$M	Phase 4 Capital US$M	Sustaining Capital US$M	Total US$M
Underground Mining				
Underground Mining	543	684	2,747	3,974
Mining Mobile Equipment	63	66	1,238	1,367
Subtotal	607	750	3,984	5,341
Power and Smelter				
Smelter Total	906	–	165	1,071
Power Infrastructure	84	134	–	218
Subtotal	990	134	165	1,289
Concentrator and Tailings				
Process Plant	262	238	193	693
Tailings	57	–	404	461
Subtotal	320	238	597	1,154
Infrastructure				
General Surface Infrastructure	662	98	150	910
Rail Spur	–	84	70	154
Subtotal	662	182	220	1,064
Indirects				
EPCM	127	141	5	273
Owners Cost	83	–	15	98
Customs Duties	92	44	175	311
Closure	–	–	145	145
Subtotal	302	185	340	826
Capital Expenditure Before Contingency	2,880	1,488	5,306	9,674
Contingency	157	65	277	499
Capital Expenditure After Contingency	3,037	1,553	5,583	10,173

Totals have been rounded.

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$6.00/lb. Cost sensitivity is shown in Table 22.9.

Table 22.8 Kamoa-Kakula 2023 PFS Mine Copper Price Sensitivity

After-Tax NPV (US$M)	Copper Price (US$/lb)									
Discount Rate	2.00	2.50	3.00	3.50	3.70	4.00	4.25	4.50	5.00	6.00
Undiscounted	12,760	23,279	33,732	43,902	47,969	54,069	59,153	64,237	72,562	87,982
4.0%	9,004	14,953	20,846	26,646	28,966	32,446	35,346	38,246	42,990	51,776
6.0%	7,734	12,363	16,940	21,463	23,272	25,986	28,248	30,509	34,211	41,069
8.0%	6,733	10,407	14,032	17,625	19,062	21,218	23,015	24,811	27,756	33,213
10.0%	5,934	8,900	11,821	14,723	15,884	17,626	19,077	20,528	22,910	27,328
12.0%	5,285	7,717	10,107	12,486	13,438	14,865	16,055	17,244	19,201	22,833
15.0%	4,519	6,369	8,183	9,992	10,715	11,800	12,704	13,609	15,101	17,875

Note: The copper price used in the economic analysis is $3.80/lb. in 2023, $3.90/lb. in 2024, $4.00/lb. in 2025, $4.00/lb. in 2026 and a long-term copper price of $3.70/lb. from 2027 onwards.

Table 22.9 Kamoa-Kakula 2023 PFS Mine Cost Sensitivity

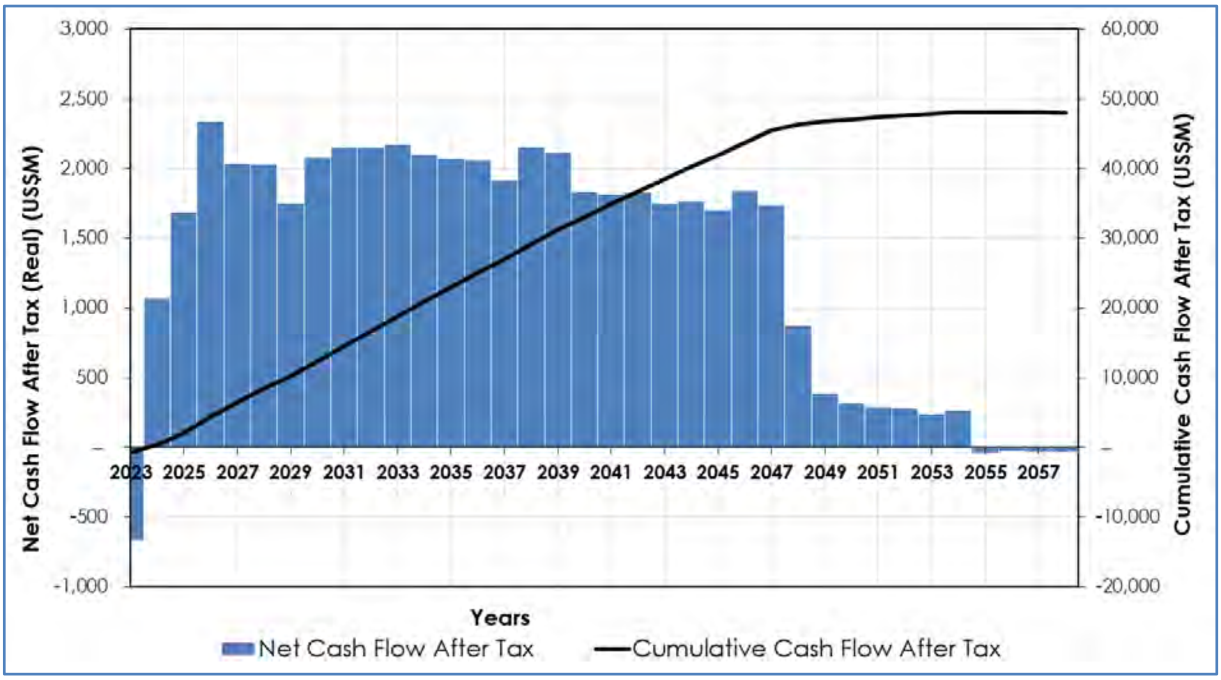
			Change from Base NPV8% (US$M)				
Variable	Units	Base Value	–25%	–10%	–	10%	25%
Initial Capital Cost	US$M	3,037	19,624	19,279	19,062	18,846	18,521
Expansion Capital Cost	US$M	1,553	19,280	19,149	19,062	18,975	18,845
Initial and Expansion Capital Cost	US$M	4,590	19,842	19,366	19,062	18,759	18,304
Site Operating Cost	US$/t Milled	78	21,163	19,903	19,062	18,221	16,959
Treatment and Refining	US$/t and US$/lb Cu	88/0.09/175	19,325	19,167	19,062	18,957	18,800
Transport	US$/t Conc	395 / 395	19,672	19,306	19,062	18,818	18,453

22.2.4 Project Cash Flow

The annual and cumulative cash flows are shown in Figure 22.5 (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 22.5 Kamoa-Kakula 2023 PFS Mine Projected Cumulative Cash Flow



OreWin, 2023.

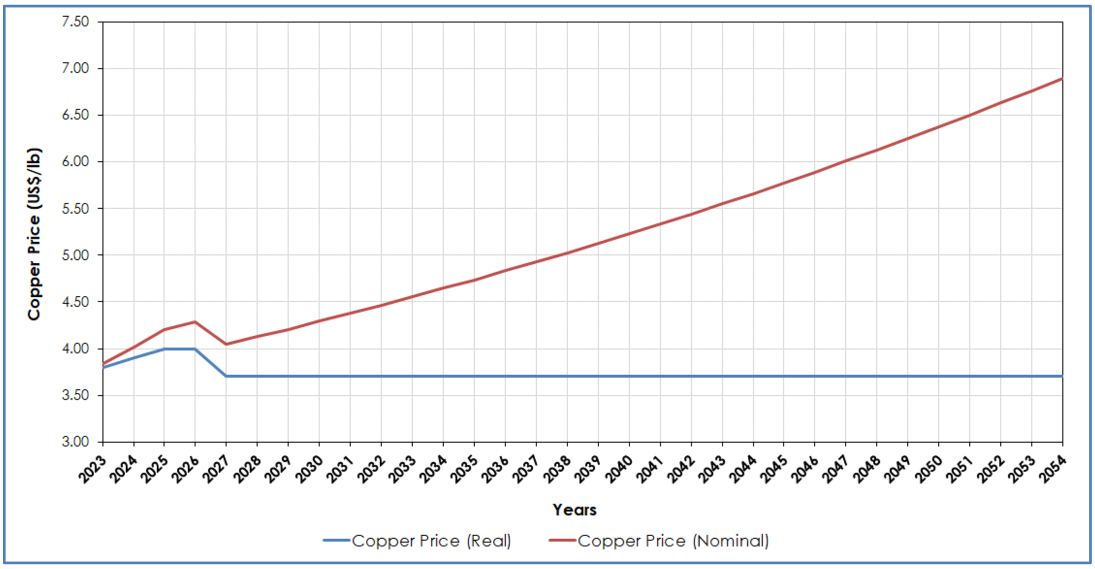
Table 22.10 Kamoa-Kakula 2023 PFS Cash Flow

Cash Flow Statement (US$M)	Year							
Year Number	Total	2023	2024	2025	2026	2027	2028	2031	2041
Year To							2030	2040	LOM
Revenue									
Copper in Blister	110,500	541	548	4,943	4,943	4,572	13,717	45,300	35,935
Copper in Concentrate	20,569	2,904	3,490	565	524	876	2,043	6,464	3,703
Acid Production	2,618	–	–	88	87	95	313	1,186	849
Gross Sales Revenue	133,687	3,445	4,038	5,596	5,554	5,543	16,074	52,950	40,486
Less: Realisation Costs									
Logistics	9,053	458	537	344	317	370	1,030	3,418	2,579
Treatment, refining and smelter charges	4,719	213	244	168	166	184	525	1,735	1,485
Royalties and Export Tax	7,417	261	308	295	292	306	869	2,881	2,206
Total Realisation Costs	21,189	932	1,090	806	775	859	2,424	8,034	6,269
Net Sales Revenue	112,498	2,513	2,948	4,790	4,779	4,684	13,650	44,917	34,218
Site Operating Costs									
UG Mining	19,380	401	471	529	551	569	1,916	7,891	7,052
Processing	7,167	144	163	211	210	211	643	2,799	2,785
Smelter	5,298	–	–	215	232	233	656	2,222	1,740
General and Administration	2,800	124	128	151	151	149	368	802	927
Total	34,644	668	763	1,106	1,144	1,162	3,583	13,715	12,505
EBITDA	77,854	1,844	2,186	3,684	3,635	3,522	10,067	31,202	21,713
EBITDA Margin	58.24%	53.53%	54.13%	65.86%	65.45%	63.55%	62.63%	58.93%	53.63%
Capital Costs									
Initial Capital	3,037	1,899	630	489	19	–	–	–	–
Expansion Capital	1,553	–	51	298	245	222	737	–	–
Sustaining Capital	5,583	177	79	354	248	334	979	2,145	1,266
Working Capital and VAT	–	–258	–45	–121	3	1	13	38	368
VAT	285	105	191	–	–	–	–	–	–11
Net Cash Flow Before-Tax	67,966	–385	1,572	2,422	3,125	2,968	8,364	29,095	20,804
Income Tax	19,997	286	504	734	789	930	2,502	8,368	5,884
Net Cash Flow After-Tax	47,969	–671	1,068	1,688	2,337	2,038	5,862	20,727	14,920

22.2.4.1 Project Cash Flow Nominal

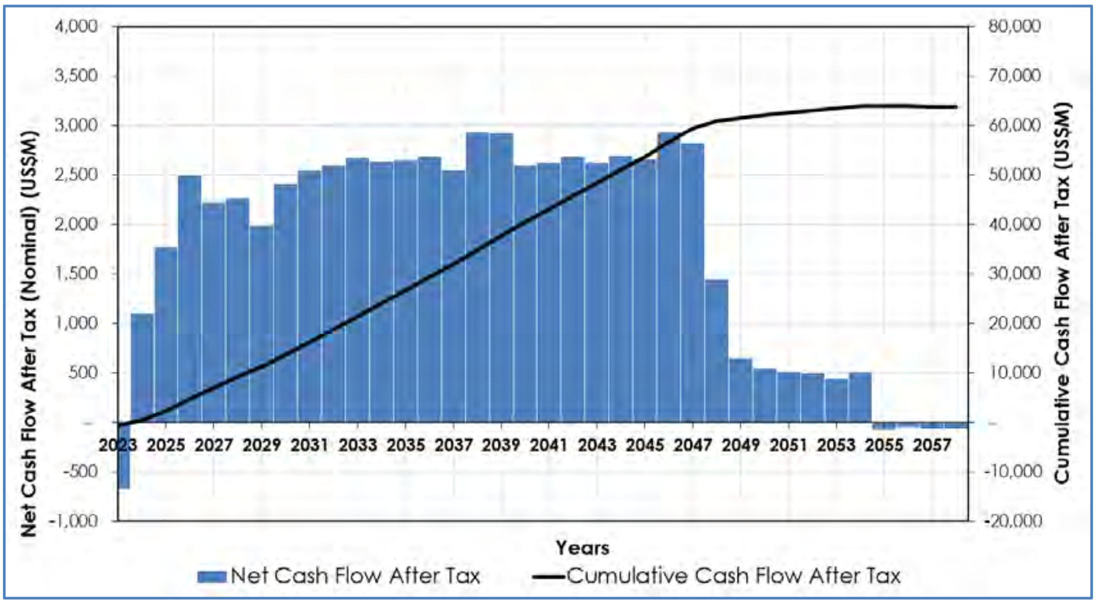
Figure 22.6 shows the Copper price assumptions in real and nominal terms and projected annual and cumulative cash flow for the Kamoa-Kakula 2023 PFS on a nominal basis, using a fixed U.S. annual inflation rate of 2%. Figure 22.7 and Table 22.11 show the Kamoa-Kakula 2023 PFS projected annual and cumulative cash flow shown on a nominal basis.

Figure 22.6 Copper Price Assumptions Shown on a Real and Nominal Basis



OreWin, 2023.

Figure 22.7 Kamoa-Kakula 2023 PFS Projected Cumulative Cash Flow Nominal Basis



OreWin, 2023.

OreWin

Table 22.11 Kamoa-Kakula 2023 PFS Cash Flow (Nominal)

Cash Flow Statement (US$M)	Year								
Year Number	Total	2023	2024	2025	2026	2027	2028	2031	2041
Year To							2030	2040	LOM
Gross Revenue	176,032	3,479	4,160	5,880	5,953	6,060	18,279	68,488	63,733
Realisation Costs	27,846	942	1,123	847	831	939	2,756	10,368	10,041
Net Revenue	148,186	2,538	3,037	5,033	5,122	5,121	15,523	58,120	53,692
Operating Costs									
Mining	25,505	399	480	549	585	617	2,180	9,905	10,790
Processing	9,702	158	183	245	249	255	814	3,506	4,292
Tailings	44	1	1	1	1	1	3	12	23
Smelter	6,956	–	–	237	263	269	791	2,792	2,604
General and Administration	3,590	121	128	154	157	157	405	1,004	1,464
Discount on Power	-528	-26	-32	-63	-69	-72	-253	-14	–
Customs (OPEX)	1,488	22	26	39	41	42	136	560	623
Total Operating Costs	46,758	675	786	1,162	1,226	1,270	4,078	17,766	19,795
EBITDA	101,429	1,862	2,252	3,871	3,896	3,851	11,446	40,354	33,897
Capital Costs									
Initial Capital	3,101	1,918	649	514	21	–	–	–	–
Expansion Capital	1,708	–	53	313	263	242	837	–	–
Sustaining Capital	7,161	179	81	372	266	365	1,111	2,751	2,035
Working Capital and VAT	266	-260	-46	-127	3	1	14	54	627
VAT	282	106	196	–	–	–	–	–	-21
Net Cash Flow Before-Tax	90,006	-389	1,619	2,545	3,349	3,245	9,511	37,656	32,468
Income Tax	26,296	289	519	771	845	1,017	2,842	10,848	9,164
Net Cash Flow After-Tax	63,710	-677	1,100	1,774	2,504	2,228	6,669	26,808	23,304

23 ADJACENT PROPERTIES

There are no adjacent properties relevant to this Report.

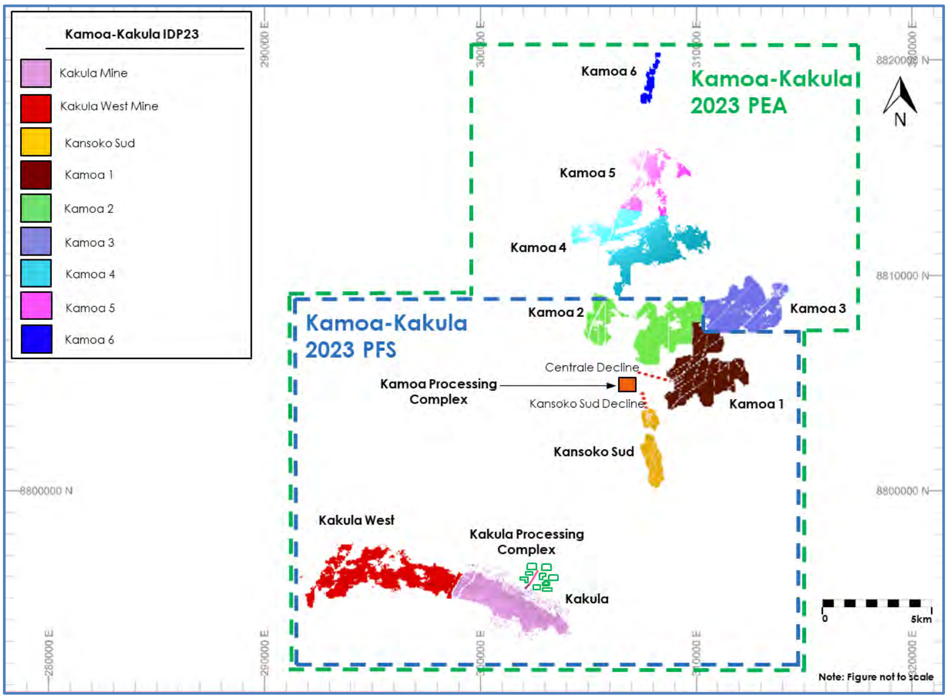
24 OTHER RELEVANT DATA AND INFORMATION

24.1 Kamoa-Kakula 2023 PEA

The Kamoa-Kakula 2023 PEA analyses a production case of maintaining overall production rate of up to 19.2 Mtpa from the Kamoa-Kakula 2023 PFS with the addition of four new underground mines in the Kamoa area (called, Kamoa 3, 4, 5, and 6). The additional four mines will extend the Kamoa-Kakula Copper Complex mine life by nine-years. Overview of deposits included within the Kamoa-Kakula 2023 PFS (outlined by blue dotted line) and the Kamoa-Kakula 2023 PEA (outlined by green dotted line) is shown in Figure 24.1. The nine mines are listed below:

- Kakula Mine (PFS 9.2 Mtpa).
- Kakula West Mine (PFS 6.2 Mtpa).
- Kamoa 1 Mine (PFS 6.0 Mtpa).
- Kansoko Sud Mine (PFS 2.0 Mtpa).
- Kamoa 2 (PFS 6.0 Mtpa).
- Kamoa 3 (PEA 6.0 Mtpa).
- Kamoa 4 (PEA 6.0 Mtpa).
- Kamoa 5 (PEA 3.0 Mtpa).
- Kamoa 6 (PEA 1.0 Mtpa).

Figure 24.1 Kamoa-Kakula 2023 PEA Mining Locations



OreWin, 2023.

The Kamoa-Kakula 2023 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 3, Kamoa 4, Kamoa 5, and Kamoa 6 for the PEA analyses have been prepared using the Mineral Resources stated in the Kamoa-Kakula 2023 Resource Update.

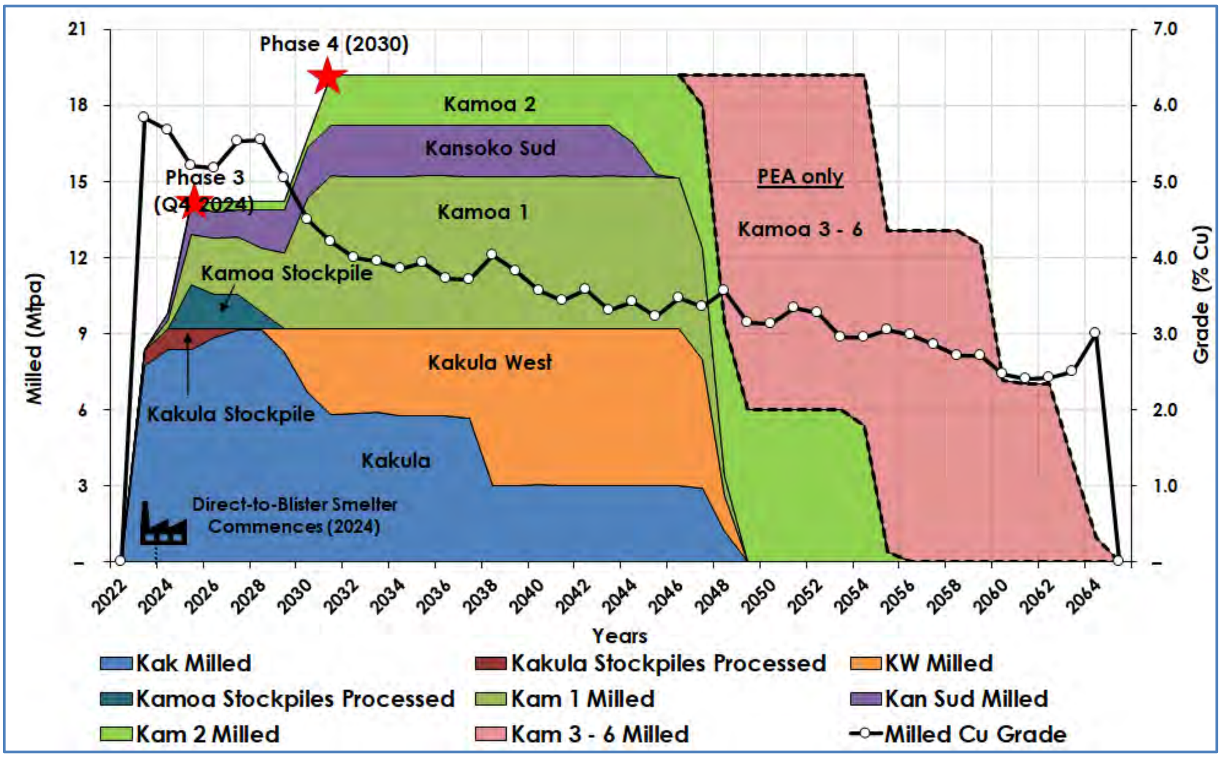
The potential development scenarios at the Kamoa-Kakula Project include the Kamoa-Kakula IDP23 development scenario shown in Figure 24.2.

The Kamoa-Kakula 2023 PEA assesses a nine-year mine life extension of the Kamoa-Kakula Copper Complex, in addition to the Kamoa-Kakula 2023 PFS. This case includes the addition of four new underground mines in the Kamoa area (called Kamoa 3, 4, 5, and 6), and additional capital, and operating costs, to bring plant feed from the Kamoa mines to the Kakula concentrator by overland conveyors. This maintains the overall production rate of up to 19.2 Mtpa.

As the resources at the Kakula, Kakula West, Kamoa 1, Kansoko Sud, and Kamoa 2 mines are mined out, production would begin sequentially at four other mines in the Kamoa complex to maintain throughput of 19.2 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 Kamoa. Included in this scenario is the construction of an overland conveyor over the years 2045 and 2046, capable of conveying required material from the Kamoa mines to the Kakula processing complex.

Figure 24.2 Kamoa-Kakula 2023 PEA Long-Term Development Scenario



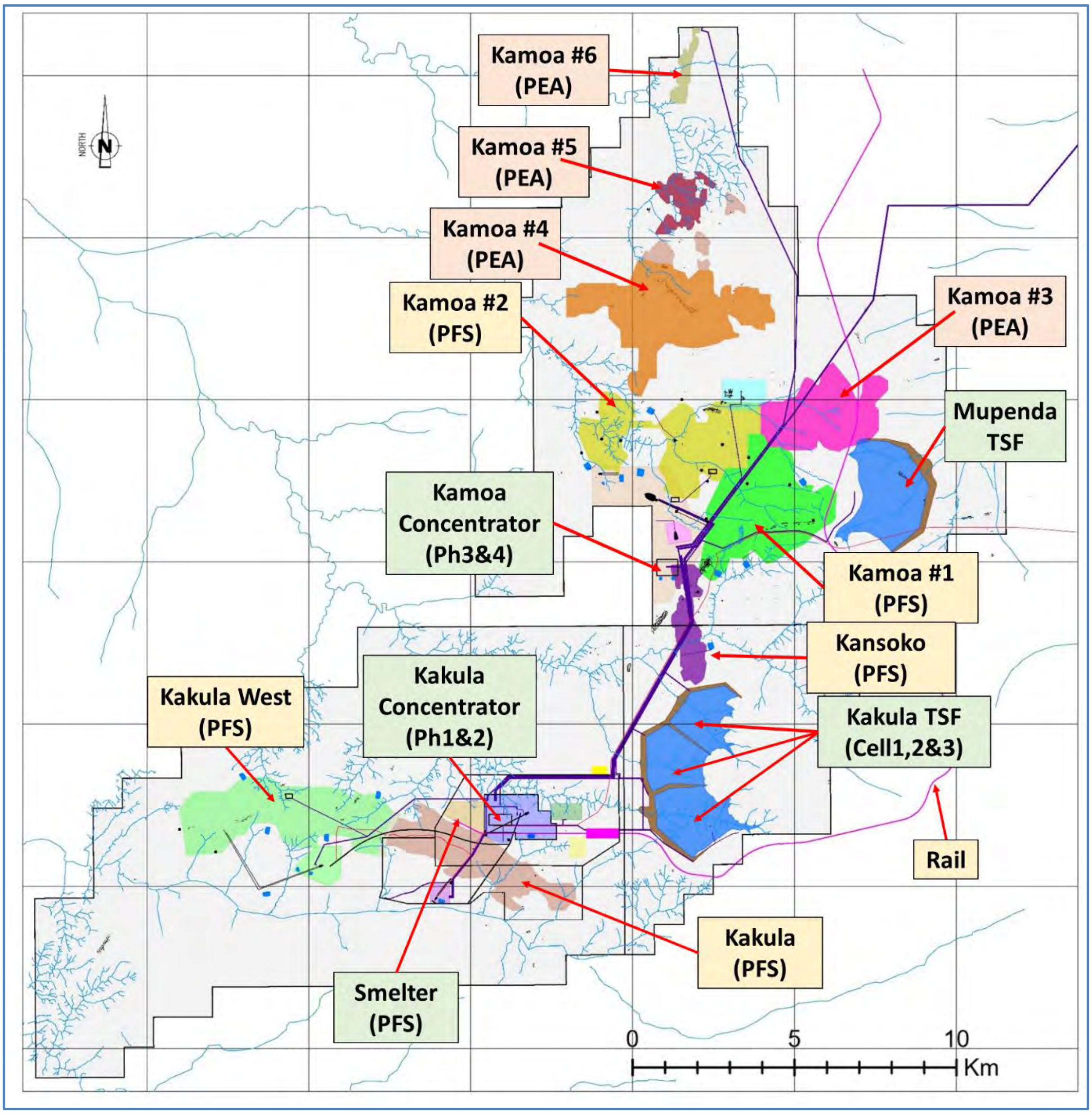
OreWin, 2023.

A site plan showing the locations of the mines and key infrastructure for Kamoa-Kakula mines is shown in Figure 24.3.

The Kamoa-Kakula 2023 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 2023 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 2023 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.

Figure 24.3 Kamoa-Kakula IDP23 Site Plan



Kamoa Copper SA, 2023.

24.2 Kamoa-Kakula 2023 PEA Assumptions

24.2.1 Economic Assumptions

24.2.1.1 Pricing and Discount Rate Assumptions

The Project level valuation model begins on 1 January 2023. It is presented in Q1'23 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$3.70/lb copper. This is reasonable based on industry forecasts, and prices used in other studies. The product being sold is copper concentrate and blister copper, and payment terms for the copper assume that the LOM average payable copper concentrate and blister copper is 96.75% and 99.70% respectively.

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 and blister production from internal and external smelter (tolling in-country). 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.

The mineral products are assumed to be sold on a Free Carrier (FCA) basis. Therefore, Kamoa Copper would not incur transportation/freight costs, as the buyer would pay for these costs directly.

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 with the use of the carried forward losses limited to 60% of the tax result of the considered year. The aggregate exploration expenditure may be claimed.

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.

Export Tax on Concentrate

A tax on the export of concentrate is levied on a per tonne basis and equates to US$100/t NSR 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 and blister product 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 moveable 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.

Tax on Excess Profits

A special tax on excess profits applies when prevailing commodity prices are more than 25% higher than those prices used in a feasibility study filed with the DRC tax authorities. A tax of 50% is levied on such incremental profits, from which income tax payments are deductible.

24.2.1.5 Sunk Costs

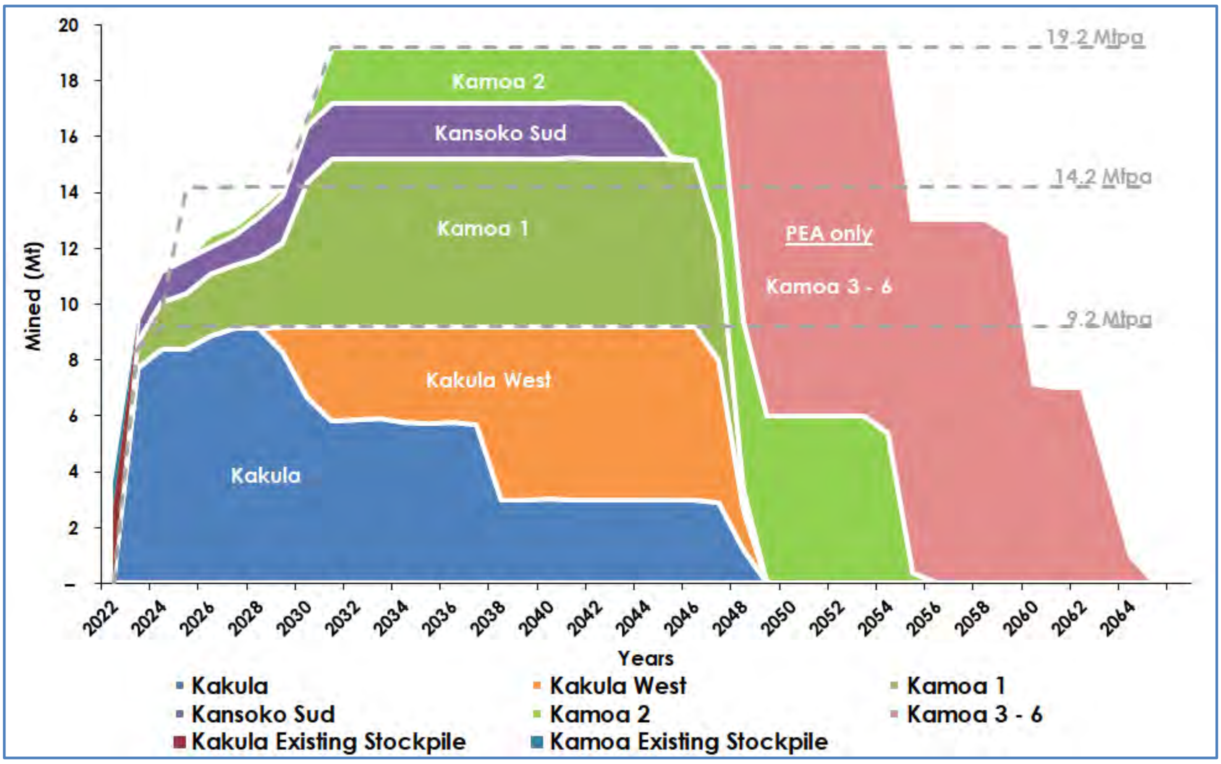
The estimate excludes all sunk costs up to 31 December 2022.

24.3 Kamoa-Kakula 2023 PEA Results

The Kamoa-Kakula 2023 PFS analyses a development option of mining several deposits (Kakula, Kakula West, Kamoa 1, Kansoko Sud, and Kamoa 2) on the Kamoa-Kakula Project as an integrated, 19.2 Mtpa mining, processing and smelting complex. Kamoa-Kakula 2023 PEA scenario envisages a nine-year mine life extension of the Kamoa-Kakula Copper Complex by the construction of four additional mines named Kamoa 3, Kamoa 4, Kamoa 5, and Kamoa 6. As the resources at the Kakula, Kakula West, Kamoa 1, Kansoko Sud, and Kamoa 2 mines are mined out, production would begin sequentially at the four other mines in the Kamoa complex to maintain throughput of 19.2 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 Kamoa. Included in this scenario is the construction of an overland conveyor over the years 2045 and 2046, capable of conveying required material from the Kamoa mines to the Kakula processing complex. The development scenario of the Kamoa-Kakula 2023 PEA is shown in Figure 24.4.

Figure 24.4 Kamoa-Kakula 2023 PEA Development Scenario



OreWin, 2023.

A summary of the key results for the Kamoa-Kakula 2023 PEA scenario are:

- Very-high-grade initial phase of production is projected to have a grade of 5.8% copper in the first year and an average grade of 4.94% copper over the initial 10-years period, resulting in estimated average annual copper production of 612,448 tonnes.
- Initial capital cost, including contingency, is estimated at US$3,037M.
- Average total cash cost of US$1.44/lb of copper during the first 10-years, including sulfuric acid credits.
- After-tax NPV, at an 8% discount rate of US$20.22b.
- After-tax payback period of 1.6-years.

The LOM production scenario provides for 653.5 Mt to be mined at an average grade of 3.70% copper, producing 51 Mt of high-grade copper concentrate, containing approximately 45.7 b pounds of copper.

The economic analysis uses a long-term price assumption of US$3.70/lb of copper and returns an after-tax NPV at an 8% discount rate of US$20.2b. It has an after-tax payback period of 1.6-years.

The estimated initial capital cost, including contingency, is US$3,037M. The capital expenditure for power infrastructure, which is included in the initial capital cost, includes a US$131M to rehabilitate the turbine #5 of the existing Inga II hydropower facility on behalf of SNEL to provide the Kamoa-Kakula Project with access to clean hydroelectricity for its planned operations. The work is being led by Voith Hydro of Germany; the advance payment will be recovered through a reduction in the power tariff.

The Kamoa-Kakula 2023 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 2023 PEA are summarised in Table 24.2. The mining production statistics are shown in Table 24.3. The Kamoa-Kakula 2023 PEA 19.2 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 2023 PEA as part of the Kamoa-Kakula IDP23 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 IDP23 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 IDP23. 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 2023 PEA Financial Results

	Discount Rate (%)	Before Taxation	After Taxation
Net Present Value (US$M)	Undiscounted	86,453	60,760
	4.0	46,742	32,708
	6.0	36,329	25,342
	8.0	29,096	20,224
	10.0	23,899	16,544
	12.0	20,049	13,818
Internal Rate of Return	–	453.8%	199.2%
Project Payback Period (Years)	–	1.2	1.6

Table 24.2 Kamoa-Kakula 2023 PEA Results Summary for 19.2 Mtpa Production

Item	Unit	Total
Total Processed		
Quantity Milled	kt	657,428
Copper Feed Grade	%	3.70
Total Concentrate Produced		
Copper Concentrate Produced	kt (dry)	50,761
Copper Recovery	%	86.45
Copper Concentrate Grade	%	41.45
Contained Copper in Concentrate	Mlb	46,384
Contained Copper in Concentrate	kt	21,040
Annual Average (First 10-Years)		
Ore Milled	kt	14,428
Copper Feed Grade	%	4.94
Copper Concentrate Produced	kt (dry)	1,379
Contained Copper in Concentrate	Mlb	1,368
Contained Copper in Concentrate	kt	620
C1 Cash Cost	US$/lb. payable Cu	1.22
EBITDA	US$M	3,151
Key Financial Results		
Remaining Phase 3 Capital Costs	US$M	3,037
Phase 4 Capital Costs Capital Costs	US$M	1,553
Sustaining Capital Costs	US$M	8,858
LOM Average C1 Cash Cost	US$/lb Payable Cu	1.32
LOM Average Total Cash Cost	US$/lb Payable Cu	1.53
LOM Average Site Operating Cost	US$/t Milled	70.57
After-Tax NPV8%	US$M	20,224
Project Life	Years	42

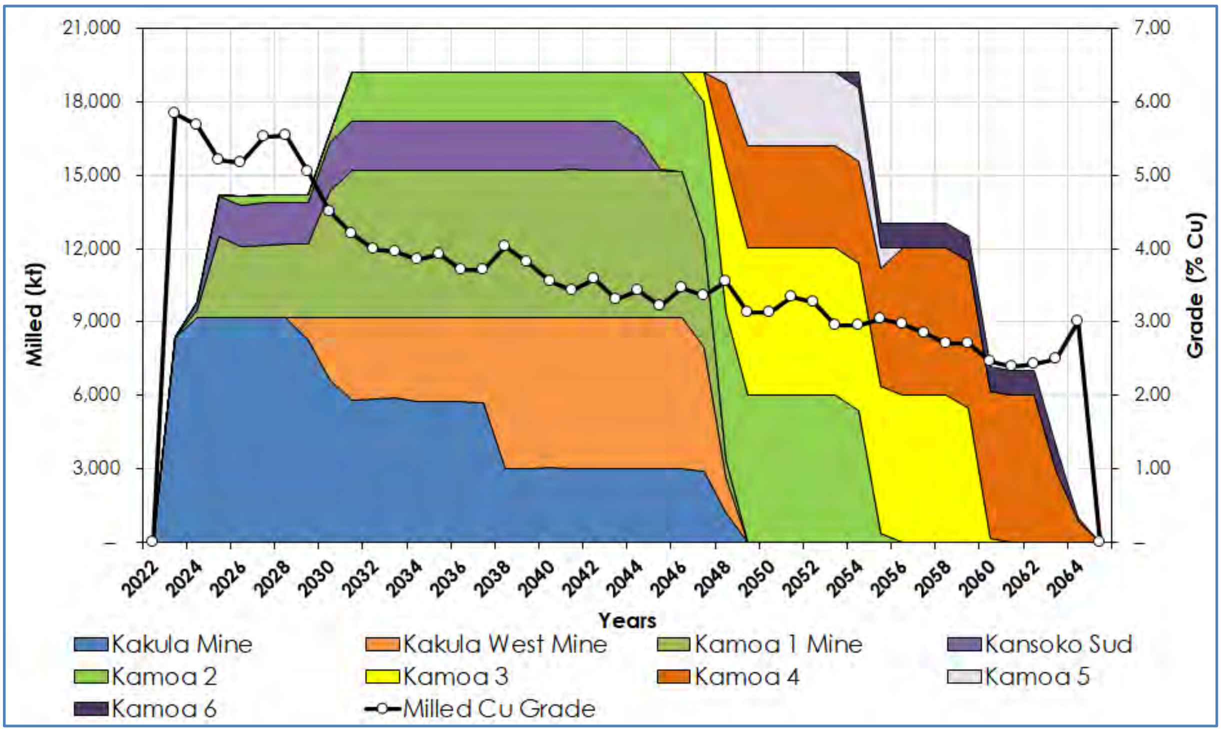
Note: The copper price used in the economic analysis is $3.80/lb. in 2023, $3.90/lb. in 2024, $4.00/lb. in 2025, $4.00/lb. in 2026 and a long-term copper price of $3.70/lb. from 2027 onwards.

Table 24.3 Kamoa-Kakula 2023 PEA Production and Processing

Item	Unit	2023-2024	2025-2029	First 10-Years	LOM Average
Total Processed					
Quantity Milled	kt	9,106	14,194	14,428	15,653
Copper Feed Grade	%	5.75	5.30	4.94	3.70
Total Concentrate Produced					
Copper Concentrate Produced	kt (dry)	917	1,431	1,379	1,209
Copper Recover	%	86.97	87.02	87.02	86.45
Copper Concentrate Grade	%	49.67	45.70	45.01	41.45
Contained Copper in Concentrate					
Contained Copper	Mlb	1,004	1,442	1,368	1,104
Contained Copper	kt	455	654	620	501
Concentrate Smelted / Sold					
Concentrate Smelted (Kamoa)	kt (dry)	–	1,133	936	945
Concentrate Tolled (LCS)	kt (dry)	134	134	134	106
Concentrate Sold	kt (dry)	783	164	310	158
Payable Copper Sold					
Blister Anodes (Kamoa)	kt	–	496	396	374
Blister Copper (LCS)	kt	64	65	63	47
Copper in Concentrate	kt	376	80	147	69
Total Payable Copper Sold					
Copper	Mlb	971	1,411	1,336	1,081
Copper	kt	440	640	606	490

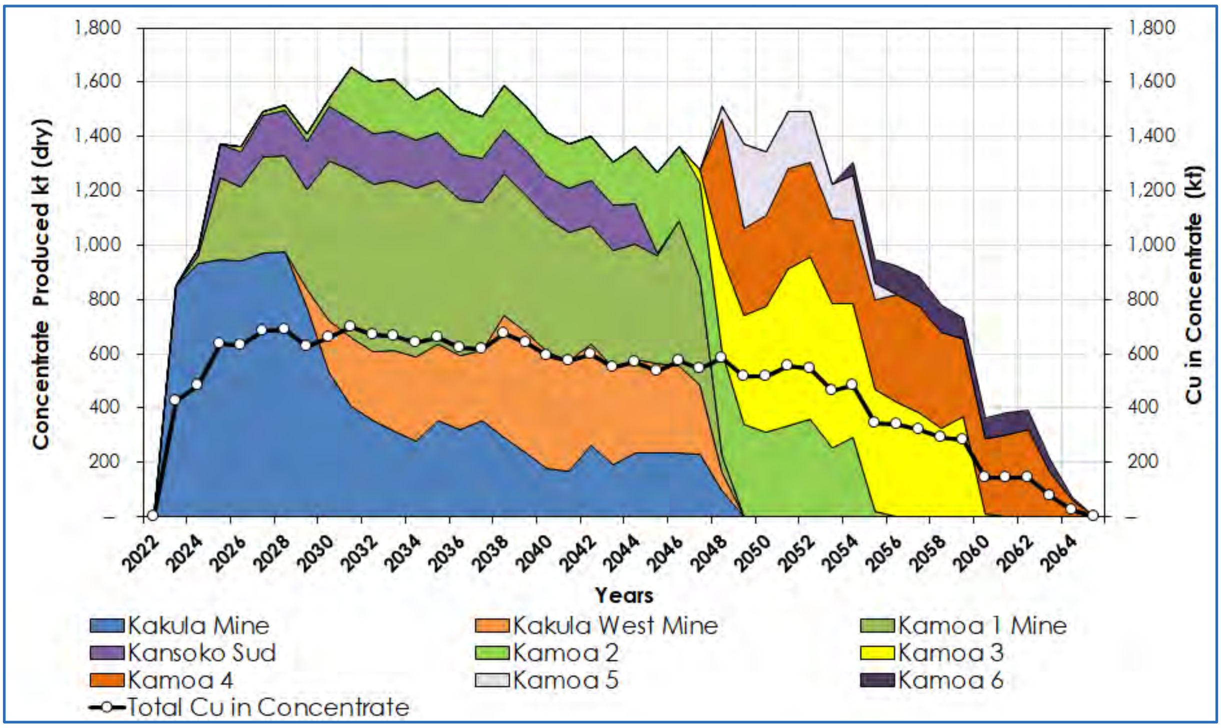
Note: The 2023-2024 average includes approximately 20 kt of copper in concentrate that is processed by the Phase 3 concentrator during the ramp-up period in 2024.

Figure 24.5 Kamoa-Kakula 2023 PEA Process Production



OreWin, 2023.

Figure 24.6 Kamoa-Kakula 2023 PEA Concentrate and Metal Production



OreWin, 2023.

The Kamoa-Kakula 2023 PEA as part of the Kamoa-Kakula IDP23 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 IDP23 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 IDP23. 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.

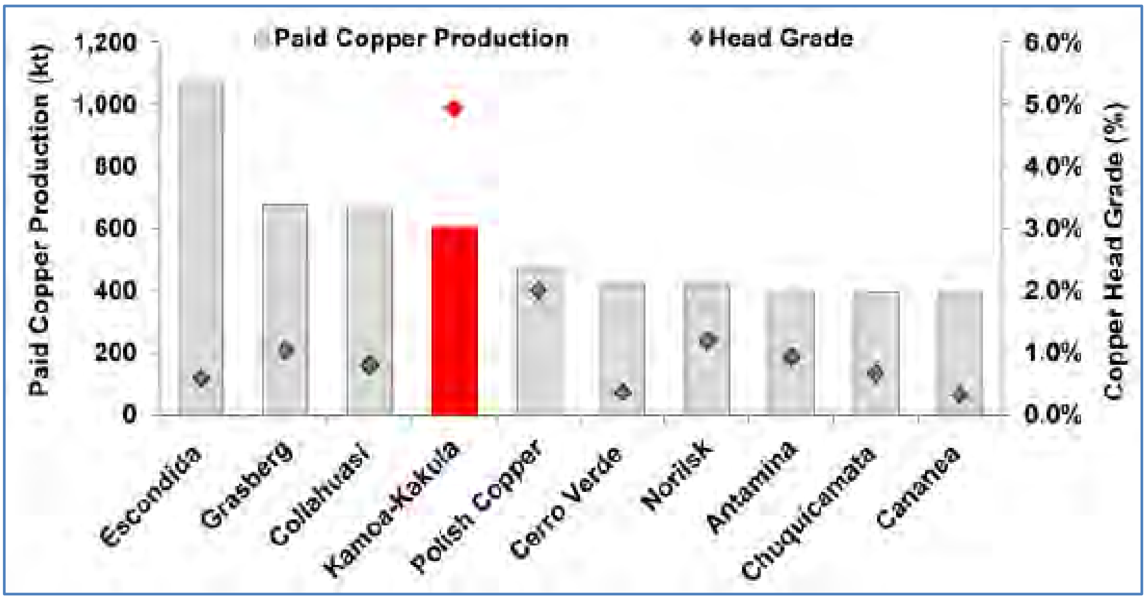
Table 24.4 summarises unit operating costs. Figure 24.7 shows the projected top 10 copper mines ranked by paid copper production (with the Kamoa-Kakula 2023 PFS first-10-years' average annual production of copper in concentrate considered to be its nominal copper production). The Kamoa-Kakula IDP23 was not reviewed by Wood Mackenzie prior to filing.

Table 24.4 Kamoa-Kakula 2023 PEA Unit Operating Costs

	Payable Copper (US$/lb)			
	2023-2024	2025-2029	First 10-Years	LOM Average
Mining	0.41	0.44	0.47	0.58
Processing	0.16	0.15	0.16	0.21
Smelter	–	0.16	0.13	0.17
Logistics	0.51	0.24	0.29	0.26
Treatment, refining and smelter charges	0.24	0.12	0.14	0.13
General and Administration	0.13	0.10	0.09	0.07
Sulfuric Acid Credits[1]	–	-0.07	-0.06	-0.09
C1 Cash Cost	1.45	1.15	1.22	1.32
Royalties and Export Tax	0.29	0.21	0.22	0.21
Total Cash Cost	1.74	1.36	1.44	1.53

Note: C1 cash costs in this table include the impact of accounting adjustments related to the addition or depletion of the surface stockpiles where applicable.
[1]Acid Selling Price $150/ t Acid.

Figure 24.7 World Copper Producer Copper Production and Head Grade



Ivanhoe, 2023. 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 2023 PEA Revenue and Operating Costs

	Total LOM (US$M)	2023-2024	2025-2029	First 10-Years	LOM Average
		(US$/t) Milled			
Revenue					
Copper in Blister	145,091	59.80	332.58	265.16	220.70
Copper in Concentrate	23,967	351.09	46.87	85.29	36.46
Acid Production	3,912	–	6.57	5.68	5.95
Gross Sales Revenue	172,970	410.89	386.02	356.13	263.10
Less: Realisation Costs					
Logistics	11,585	54.68	24.19	26.57	17.62
Treatment, refining and smelter charges	5,816	25.10	12.23	13.00	8.85
Royalties and Export Tax	9,572	31.25	20.74	20.53	14.56
Total Realisation Costs	26,972	111.02	57.15	60.10	41.03
Net Sales Revenue	145,998	299.87	328.87	296.02	222.07
Site Operating Costs					
Underground Mining	26,274	47.90	40.19	41.85	39.97
Processing	9,409	16.86	14.72	14.86	14.31
Smelter	7,724	–	15.79	12.32	11.75
General and Administration	2,985	13.83	10.02	8.63	4.54
Total	46,393	78.58	80.72	77.66	70.57
EBITDA	99,606	221.29	248.15	218.36	151.51
EBITDA Margin (%)	57.6%	53.86%	64.28%	61.31%	57.59%

Note: The copper price used in the economic analysis is $3.80/lb. in 2023, $3.90/lb. in 2024, $4.00/lb. in 2025, $4.00/lb. in 2026 and a long-term copper price of $3.70/lb. from 2027 onwards.

Table 24.6 Kamoa-Kakula 2023 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	543	684	4,922	6,149
Mining Mobile Equipment	63	66	1,735	1,864
Subtotal	607	750	6,657	8,013
Power and Smelter				
Smelter Total	906	–	215	1,121
Power Infrastructure	84	134	–	218
Subtotal	990	134	215	1,339
Concentrator and Tailings				
Process Plant	262	238	307	807
Tailings	57	–	534	591
Subtotal	320	238	841	1,398
Infrastructure				
General Surface Infrastructure	662	98	236	997
Rail Spur	–	84	95	179
Subtotal	662	182	406	1,250
Indirects				
EPCM	127	141	5	273
Owners Cost	83	–	15	98
Customs Duties	92	44	260	396
Closure	–	–	145	145
Subtotal	302	185	425	912
Capital Expenditure Before Contingency	2,880	1,488	8,544	12,912
Contingency	157	65	314	536
Capital Expenditure After Contingency	3,037	1,553	8,858	13,448

Note: The remaining Phase 3 capital cost of $3,037 million includes approximately $2,529 million that will be spent in 2023 and 2024, before the commissioning of the Phase 3 concentrator and other infrastructure, and an additional $508 million incurred in 2025 and 2026 that is related to the completion of the ramp-up of the underground mining operations to sustain a total production rate of 14.2 Mtpa.

The after-tax NPV sensitivity to metal price variation is shown in Table 22.7 for copper prices from US$2.00–US$6.0/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 24.8 (annual cash flow is shown on the left vertical axis and cumulative cash flow on the right axis).

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Table 24.7 Kamoa-Kakula 2023 PEA Copper Price Sensitivity

After-Tax NPV (US$M)	Copper Price (US$/lb)										
Discount Rate	2.00	2.50	3.00	3.50	3.70	4.00	4.25	4.50	5.00	5.50	6.00
Undiscounted	13,765	28,112	41,731	55,323	60,760	68,915	75,710	82,506	93,703	104,102	114,500
4.0%	9,315	16,385	23,190	29,989	32,708	36,787	40,187	43,586	49,149	54,300	59,451
6.0%	7,902	13,155	18,235	23,312	25,342	28,388	30,927	33,465	37,616	41,459	45,301
8.0%	6,822	10,849	14,756	18,661	20,224	22,567	24,520	26,472	29,667	32,625	35,583
10.0%	5,980	9,149	12,231	15,312	16,544	18,393	19,934	21,474	23,998	26,337	28,675
12.0%	5,308	7,859	10,342	12,825	13,818	15,308	16,549	17,791	19,829	21,719	23,609
15.0%	4,526	6,431	8,287	10,143	10,885	11,998	12,926	13,854	15,383	16,803	18,223

Note: The copper price used in the economic analysis is $3.80/lb. in 2023, $3.90/lb. in 2024, $4.00/lb. in 2025, $4.00/lb. in 2026 and a long-term copper price of $3.70/lb. from 2027 onwards.

Figure 24.8 Kamoa-Kakula 2023 PEA Projected Cumulative Cash Flow

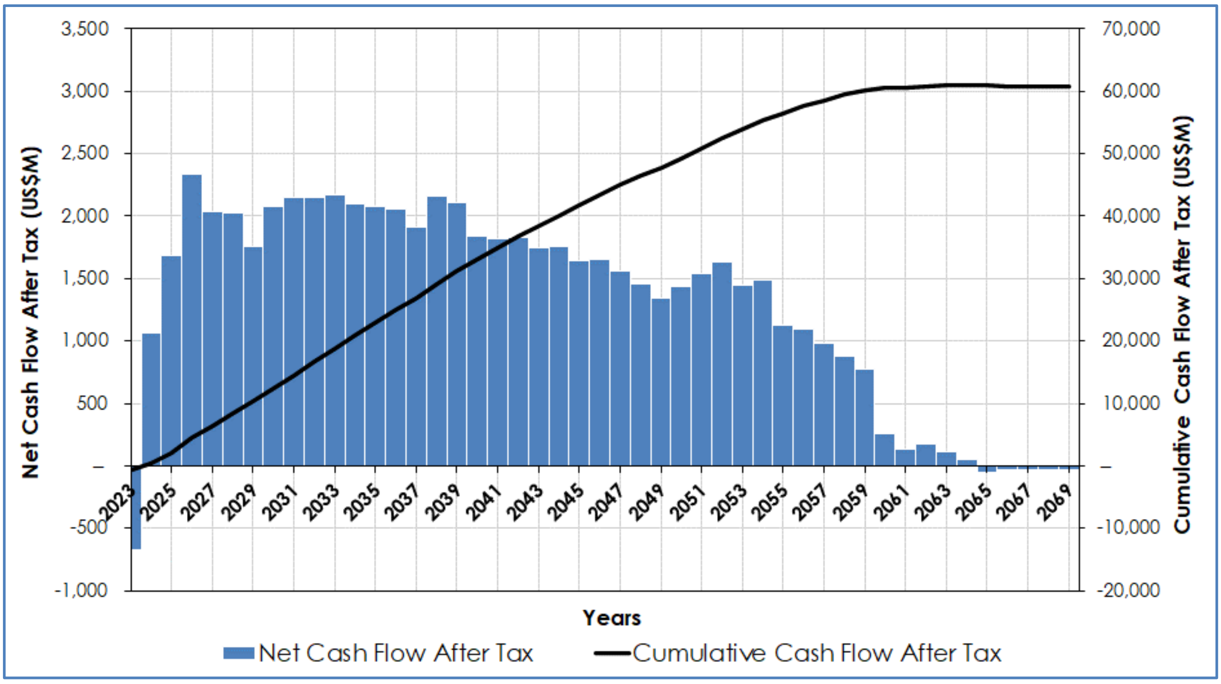

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Table 24.8 Kamoa-Kakula 2023 PEA Cash Flow

Cash Flow Statement (US$M)	Years								
Year Number	Total	2023	2024	2025	2026	2027	2028	2031	2041
Year To							2030	2040	LOM
Revenue									
Copper in Blister	145,091	541	548	4,943	4,943	4,572	13,717	45,300	70,526
Copper in Concentrate	23,967	2,904	3,490	565	524	876	2,043	6,464	7,101
Acid Production	3,912	–	–	88	87	95	313	1,186	2,142
Gross Sales Revenue	172,970	3,445	4,038	5,596	5,554	5,543	16,074	52,950	79,770
Less: Realisation Costs									
Logistics	11,585	458	537	344	317	370	1,030	3,418	5,110
Treatment, refining and smelter charges	5,816	213	244	168	166	184	525	1,735	2,582
Royalties and Export Tax	9,572	261	308	295	292	306	869	2,881	4,360
Total Realisation Costs	26,972	932	1,090	806	775	859	2,424	8,034	12,052
Net Sales Revenue	145,998	2,513	2,948	4,790	4,779	4,684	13,650	44,917	67,718
Site Operating Costs									
UG Mining	26,274	401	471	529	551	569	1,916	7,891	13,947
Processing	9,409	144	163	211	210	211	643	2,799	5,028
Smelter	7,724	–	–	215	232	233	656	2,222	4,166
General and Administration	2,985	124	128	151	151	149	368	802	1,112
Total	46,393	668	763	1,106	1,144	1,162	3,583	13,715	24,253
EBITDA	99,606	1,844	2,186	3,684	3,635	3,522	10,067	31,202	43,465
EBITDA Margin	57.59%	53.53%	54.13%	65.83%	65.45%	63.55%	62.63%	58.93%	54.49%
Capital Costs									
Initial Capital	3,037	1,899	630	489	19	–	–	–	–
Expansion Capital	1,553	–	51	298	245	222	737	–	–
Sustaining Capital	8,858	177	79	354	248	334	1,663	1,798	4,205
Working Capital and VAT	–	-258	-45	-121	3	1	7	43	369
VAT	296	105	191	–	–	–	–	–	–
Net Cash Flow Before-Tax	86,453	-385	1,572	2,422	3,125	2,968	14,241	28,375	34,134
Income Tax	25,694	286	504	734	789	930	4,079	8,306	10,06
Net Cash Flow After-Tax	60,760	-671	1,068	1,688	2,337	2,038	10,162	20,070	24,068

Table 24.9 Kamoa-Kakula 2023 PEA Processing Production Schedule

Description	Units	Totals	Years														
			2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037
Quantity Milled	kt	657,428	8,375	9,836	14,201	14,161	14,202	14,205	14,201	16,701	19,200	19,200	19,200	19,200	19,200	19,200	19,200
Cu Feed Grade	% Cu	3.70	5.83	5.68	5.20	5.17	5.52	5.54	5.05	4.49	4.21	3.99	3.95	3.85	3.93	3.72	3.71
Fe Feed Grade	% Fe	5.15	4.45	4.46	4.39	4.89	4.90	4.94	5.05	5.24	5.45	5.40	5.49	5.51	5.75	5.70	5.44
As Feed Grade	% As	00.0006	0.0001	0.0002	0.0003	0.0004	0.0005	0.0003	0.0003	0.0003	0.0004	0.0003	0.0005	0.0005	0.0006	0.0005	0.0004
S Feed Grade	% S	1.74	1.42	1.49	1.62	1.74	1.83	1.88	1.90	1.96	1.90	1.79	2.00	1.80	1.97	1.96	1.82
Copper Concentrate Produced	kt (dry)	50,761	850	983	1,374	1,362	1,492	1,516	1,412	1,541	1,652	1,604	1,611	1,535	1,578	1,502	1,472
Copper Concentrate - External Smelter	kt (dry)	6,652	717	849	132	123	222	233	112	173	296	239	233	164	203	124	102
Copper Concentrate - Internal Smelter	kt (dry)	39,672	–	–	1,108	1,106	1,136	1,150	1,166	1,235	1,223	1,232	1,244	1,237	1,241	1,245	1,236
Copper Concentrate - External Smelter (Tolling in-country)	%	4,437	134	134	134	134	134	134	134	134	134	134	134	134	134	134	134
Copper Concentrate Recovery	kt (dry)	86.45	87.03	86.91	86.44	86.43	87.08	87.48	87.63	87.69	86.44	87.05	87.73	86.95	87.41	87.40	86.96
Copper Concentrate Grade	% Cu	41.45	50.00	49.39	46.42	46.48	45.78	45.38	44.49	42.72	42.26	41.63	41.29	41.84	41.79	41.52	42.06
Contained Copper in Concentrate - External Smelter	Mlb	6,601	790	925	146	135	245	257	124	191	298	234	229	170	209	129	112
Contained Copper in Concentrate. - External Smelter	kt	2,994	358	420	66	61	111	116	56	86	135	106	104	77	95	58	51
Contained Copper in Blister - Internal Smelter	Mlb	34,778	–	–	1,097	1,097	1,097	1,097	1,097	1,097	1,090	1,090	1,090	1,090	1,091	1,091	1,091
Contained Copper in Blister - Internal Smelter	kt	15,775	–	–	497	497	497	497	497	497	495	495	495	495	495	495	495
Contained Copper in Blister - External Smelter (Tolling in-country)	Mlb	4,342	143	141	143	143	143	143	143	143	130	127	127	134	133	135	141
Contained Copper in Blister - External Smelter (Tolling in-country)	kt	1,969	65	64	65	65	65	65	65	65	59	58	58	61	60	61	64
Total Recovered Copper Production	Mlb	45,720	933	1,066	1,386	1,375	1,484	1,496	1,363	1,430	1,518	1,451	1,446	1,395	1,433	1,354	1,344
Total Recovered Copper Production	kt	20,738	423	483	628	624	673	679	618	649	689	658	656	633	650	614	610

Table 24.10 Kamoa-Kakula 2023 PEA Processing Production Schedule

Description	Units	Project Time (Years)															
		2038	2039	2040	2041	2042	2043	2044	2045	2046	2047	2048	2049	2050	2051	2052	2053
Quantity Milled	kt	19,200	19,200	19,200	19,200	19,200	19,200	19,200	19,200	19,200	19,200	19,200	19,200	19,200	19,200	19,200	19,199
Cu Feed Grade	% Cu	4.03	3.81	3.56	3.43	3.58	3.30	3.42	3.22	3.47	3.36	3.55	3.14	3.13	3.34	3.27	2.96
Fe Feed Grade	% Fe	5.31	5.41	5.36	5.57	5.46	5.39	5.22	5.26	5.28	5.36	5.44	5.48	5.22	5.21	4.97	4.90
As Feed Grade	% As	0.0003	0.0005	0.0006	0.0006	0.0006	0.0005	0.0006	0.0005	0.0004	0.0006	0.0009	0.0008	0.0008	0.0007	0.0008	0.0012
S Feed Grade	% S	1.70	1.91	1.82	2.02	1.93	1.70	1.72	1.84	1.70	1.74	1.93	1.73	1.74	1.90	1.73	1.44
Copper Concentrate Produced	kt (dry)	1,588	1,510	1,416	1,371	1,401	1,305	1,361	1,268	1,361	1,278	1,511	1,372	1,342	1,491	1,490	1,225
Copper Concentrate - External Smelter	kt (dry)	208	134	45	3	48	–	–	–	5	–	324	240	285	354	272	–
Copper Concentrate - Internal Smelter	kt (dry)	1,247	1,243	1,237	1,234	1,220	1,218	1,229	1,207	1,222	1,197	1,054	998	923	1,004	1,084	1,225
Copper Concentrate - External Smelter (Tolling in-country)	kt (dry)	134	134	134	134	134	87	132	62	134	80	134	134	134	134	134	–
Copper Concentrate Recovery	%	87.36	87.22	87.05	86.97	86.67	86.57	86.93	86.58	86.27	84.54	85.56	85.55	85.54	86.61	86.64	82.12
Copper Concentrate Grade	% Cu	42.54	42.28	41.98	41.82	42.52	42.02	41.96	42.25	42.21	42.67	38.61	37.54	38.34	37.20	36.55	38.05
Contained Copper in Concentrate - External Smelter	Mlb	229	147	49	4	53	–	–	–	6	–	298	199	241	290	219	–
Contained Copper in Concentrate - External Smelter	kt	104	67	22	2	24	–	–	–	3	–	135	90	109	132	99	–
Contained Copper in Blister - Internal Smelter	Mlb	1,097	1,097	1,097	1,097	1,097	1,097	1,097	1,097	1,097	1,097	853	814	769	811	861	1,012
Contained Copper in Blister - Internal Smelter	kt	497	497	497	497	497	497	497	497	497	497	387	369	349	368	390	459
Contained Copper in Blister - External Smelter (Tolling in-country)	Mlb	143	143	143	143	143	93	141	66	143	86	119	107	109	106	104	–
Contained Copper in Blister - External Smelter (Tolling in-country)	kt	65	65	65	65	65	42	64	30	65	39	54	49	50	48	47	–
Total Recovered Copper Production	Mlb	1,468	1,387	1,289	1,243	1,292	1,190	1,238	1,163	1,245	1,182	1,270	1,120	1,119	1,207	1,184	1,012
Total Recovered Copper Production	kt	666	629	585	564	586	540	561	527	565	536	576	508	508	548	537	459

Table 24.11 Kamoa-Kakula 2023 PEA Processing Production Schedule

Description	Units	Project Time (Years)															
		2054	2055	2056	2057	2058	2059	2060	2061	2062	2063	2064	2065	2066	2067	2068	2069
Quantity Milled	kt	19,200	13,042	13,041	13,040	13,040	12,518	7,151	7,020	7,020	3,900	974	–	–	–	–	–
Cu Feed Grade	% Cu	2.96	3.04	2.98	2.85	2.70	2.70	2.46	2.39	2.42	2.50	3.00	–	–	–	–	–
Fe Feed Grade	% Fe	4.94	4.00	4.15	4.27	4.83	5.13	5.24	4.66	3.68	3.51	5.36	–	–	–	–	–
As Feed Grade	% As	0.0009	0.0005	0.0008	0.0009	0.0012	0.0015	0.0020	0.0016	0.0013	0.0015	0.0013	–	–	–	–	–
S Feed Grade	% S	1.53	1.20	1.17	1.27	1.47	1.61	1.59	1.40	1.23	1.11	1.68	–	–	–	–	–
Copper Concentrate Produced	kt (dry)	1,305	945	923	883	781	733	363	381	393	209	71	–	–	–	–	–
Copper Concentrate - External Smelter (Tolling in-country)	kt (dry)	–	–	–	–	–	–	230	247	260	76	–					
Copper Concentrate - External Smelter	kt (dry)	1,305	945	923	883	781	733	–	–	–	–	–	–	–	–	–	–
Copper Concentrate - Internal Smelter	kt (dry)	–	–	–	–	–	–	134	134	134	134	71	–	–	–	–	–
Copper Concentrate Recovery	%	85.41	87.10	87.02	86.56	83.03	82.90	81.06	84.23	85.06	81.02	87.79	–	–	–	–	–
Copper Concentrate Grade	% Cu	37.12	36.52	36.62	36.37	37.49	38.21	39.29	37.18	36.71	37.70	35.97	–	–	–	–	–
Contained Copper in Concentrate - External Smelter	Mlb	–	–	–	–	–	–	199	203	210	63	–	–	–	–	–	–
Contained Copper in Concentrate - External Smelter	kt	–	–	–	–	–	–	90	92	95	29	–	–	–	–	–	–
Contained Copper in Blister - Internal Smelter	Mlb	1,052	749	734	697	636	608	–	–	–	–	–	–	–	–	–	–
Contained Copper in Blister - Internal Smelter	kt	477	340	333	316	288	276	–	–	–	–	–	–	–	–	–	–
Contained Copper in Blister - External Smelter (Tolling in-country)	Mlb	–	–	–	–	–	–	112	106	105	108	55					
Contained Copper in Blister - External Smelter (Tolling in-country)	kt	–	–	–	–	–	–	51	48	48	49	25					
Total Recovered Copper Production	Mlb	1,052	749	734	697	636	608	311	309	315	171	55	–	–	–	–	–
Total Recovered Copper Production	kt	477	340	333	316	288	276	141	140	143	77	25	–	–	–	–	–

24.4 Kamoa-Kakula 2023 PEA Mining

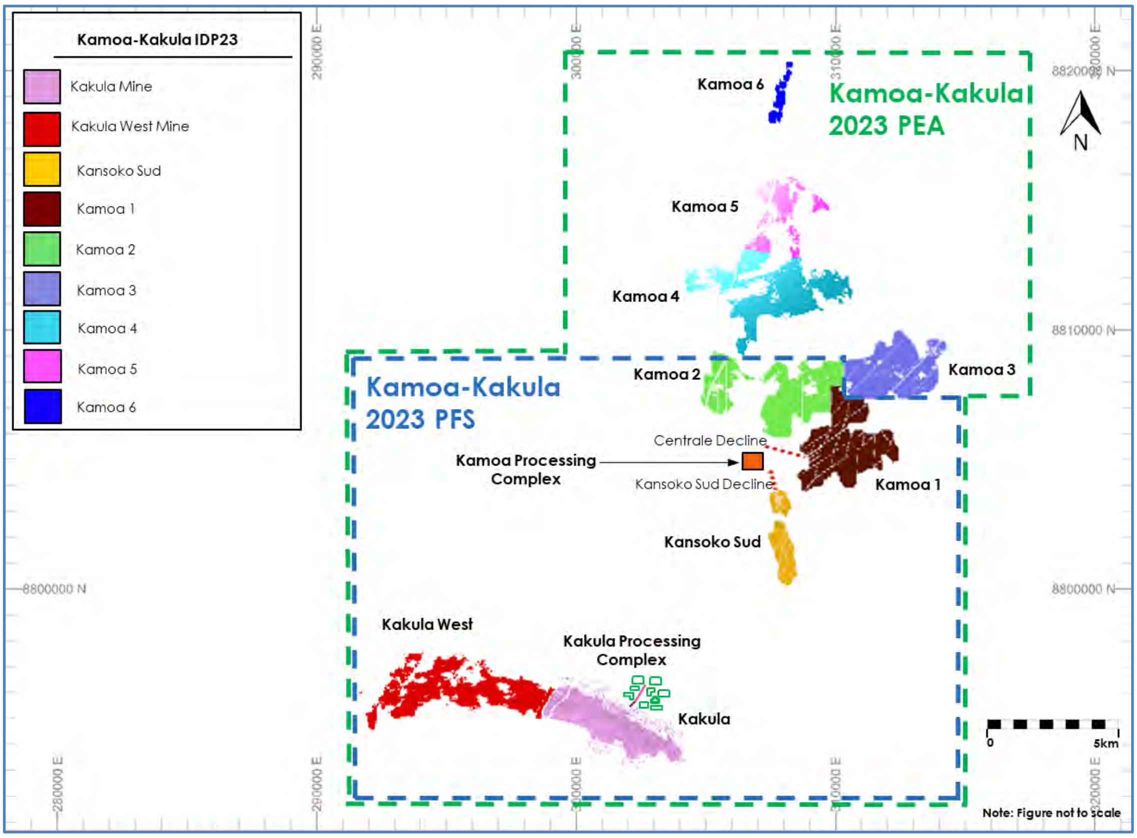
The Kamoa-Kakula 2023 PEA analyses a production case of maintaining overall production rate of up to 19.2 Mtpa from the Kamoa-Kakula 2023 PFS with the addition of four new underground mines in the Kamoa area (called, Kamoa 3, 4, 5, and 6). The additional four mines will extend the Kamoa-Kakula Copper Complex mine life by nine-years. The nine mines including four additional PEA mines are listed below:

- Kakula Mine (PFS 9.2 Mtpa).
- Kakula West Mine (PFS 6.2 Mtpa).
- Kamoa 1 Mine (PFS 6.0 Mtpa).
- Kansoko Sud Mine (PFS 2.0 Mtpa).
- Kamoa 2 (PFS 6.0 Mtpa).
- Kamoa 3 (PEA 6.0 Mtpa).
- Kamoa 4 (PEA 6.0 Mtpa).
- Kamoa 5 (PEA 3.0 Mtpa).
- Kamoa 6 (PEA 1.0 Mtpa).

Mining method in the Kamoa-Kakula 2023 PEA is assumed to be drift-and-fill with paste fill mining method.

Figure 24.9 shows the locations of the nine mines and the boundaries for the PFS and PEA cases in IDP23.

Figure 24.9 Kamoa-Kakula 2023 PEA Development and Mining Zones



OreWin, 2023.

24.4.1 Kamoa 3

Mine Planning

The preliminary mineable area was obtained using a stope shape optimiser (applied to the Kamoa 3 resource model). Stope optimisation was undertaken on the resource model at a mining cut-off grade of $100/t NSR with the outlier blocks that did not provide a consistent mineable shape removed from the targeted resource. The stope shape optimiser utilised provided assumptions to produce mineable shapes suitable for the chosen mining method.

The primary mining method for the Kamoa-Kakula deposits (drift-and-fill) was selected based on the results of a trade-off study which compared six different variations of drift-and-fill mining to identify the most suitable. The selected drift-and-fill mining method is explained in detail in Section 16.2.2.

The Kamoa 3 area dimensions are 4.06 km x 2.75 km and the depth of the Kamoa 3 targeted resource is between 45 m–1,450 m below surface. Access will be via a box-cut and twin declines on the north-west side of the Kamoa 3 deposit. The drift-and-fill mining zones will be filled and protected by paste fill. The main access to the drift-and-fill production areas is via the perimeter service access drifts and the connection drifts. The extraction ratio of the drift-and-fill zones for all lifts is 98.4%, with the mined tonnes and grades diluted by 1.8% paste fill.

Mine Access

The decline was designed to accommodate two parallel drives with a maximum gradient of ±8.5°. The declines contain a primary mine service access and a conveyor haulage drift. The service decline has dimensions of 6.0 m W x 6.0 m H, with the conveyor decline 7.5 m W x 6.0 m H.

The main conveyor decline is wide enough to accommodate the conveyor as well as large mobile equipment, 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 1,355 m from the portal, 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 shown in Figure 24.10.

From the bottom of the decline, 6 m w x 6.0 m H perimeter service drifts will be driven to the southern extremities of the deposit. This development will serve as the primary access 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 conveyor drifts are located central to the orebody. The conveyor system extends to the lower western and upper north-eastern extremities of the orebody and transports material to the main north conveyor decline. The main north decline rock handling system then transports ore to surface via the main north conveyor decline. Conveyor drifts are 7.5 m w x 6.0 m H and are driven at a maximum gradient of ±9.0°.

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

Mine Ventilation

The mine layout schematic illustrating the location of the ventilation shafts is shown in Figure 24.11. The mine will be supplied with fresh air from 4 intake ventilation shafts located central to the orebody. The ventilation air will naturally flow through the perimeter drives to the extremities of the orebody. The ventilation will be extracted via four exhaust shafts located at the extremities of the orebody. VS-01 will ventilate the conveyor belt directly to return. 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. The model shows the primary ventilation requirements of 2,500 kg/s at the peak production rate of 6.0 Mtpa and a maximum depth of 1,450 m.

The mine ventilation cooling requirements were estimated from first principles and confirmed by VentSim models which considered auto-compression, surrounding and broken rock heat, vehicle heat, and all other heat components. The model indicated that 15 MW of mechanical cooling will be required at the production rate of 6.0 Mtpa while producing at 1,347 m below surface.

For Kamoa 3, it is proposed that stand-alone refrigeration machines will serve a modular type BAC at the top of the new intake ventilation shafts at VS-04, VS-06 and VS-08. Each installation will provide a nominal cooling duty of 5MWR to underground. 5.5MWR of refrigeration machine duty per installation will be required (including cooling losses). The mechanical refrigeration modules will be phased-in as required.

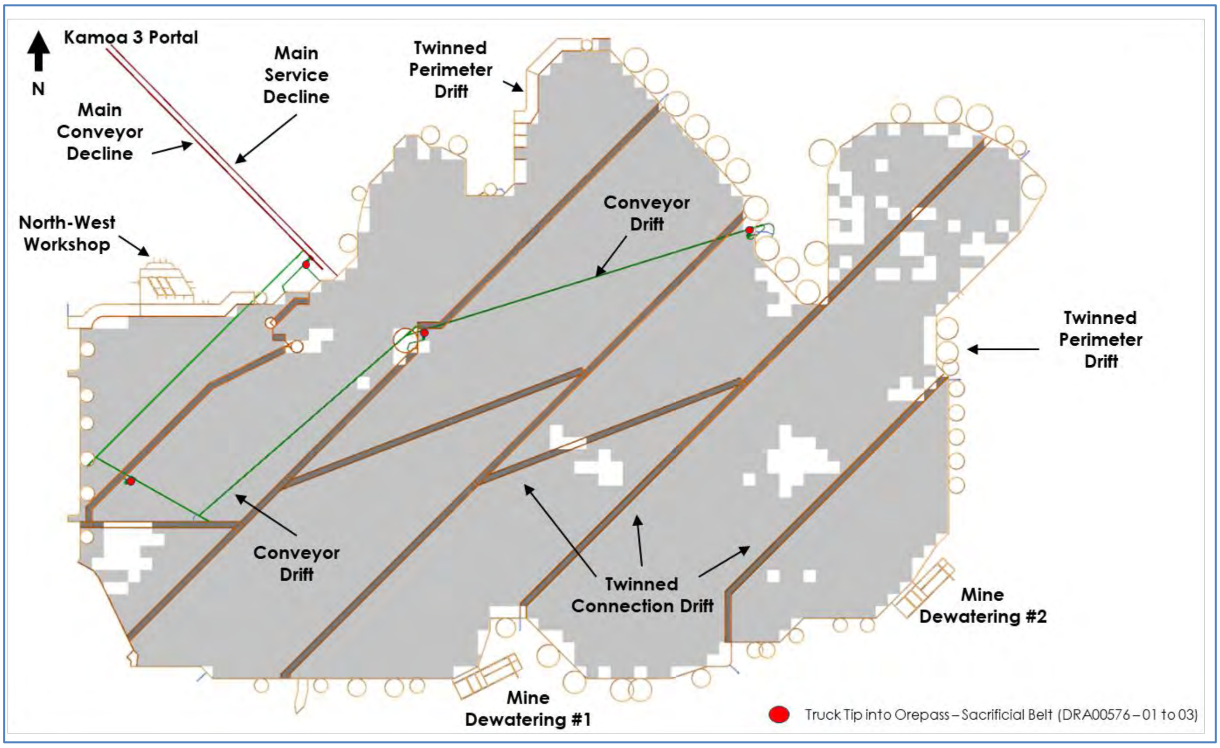
Backfill

The Backfill boreholes were strategically designed and located to supply backfill to the drift-and-fill panels over the life of the project.

Mine Development and Production Schedule

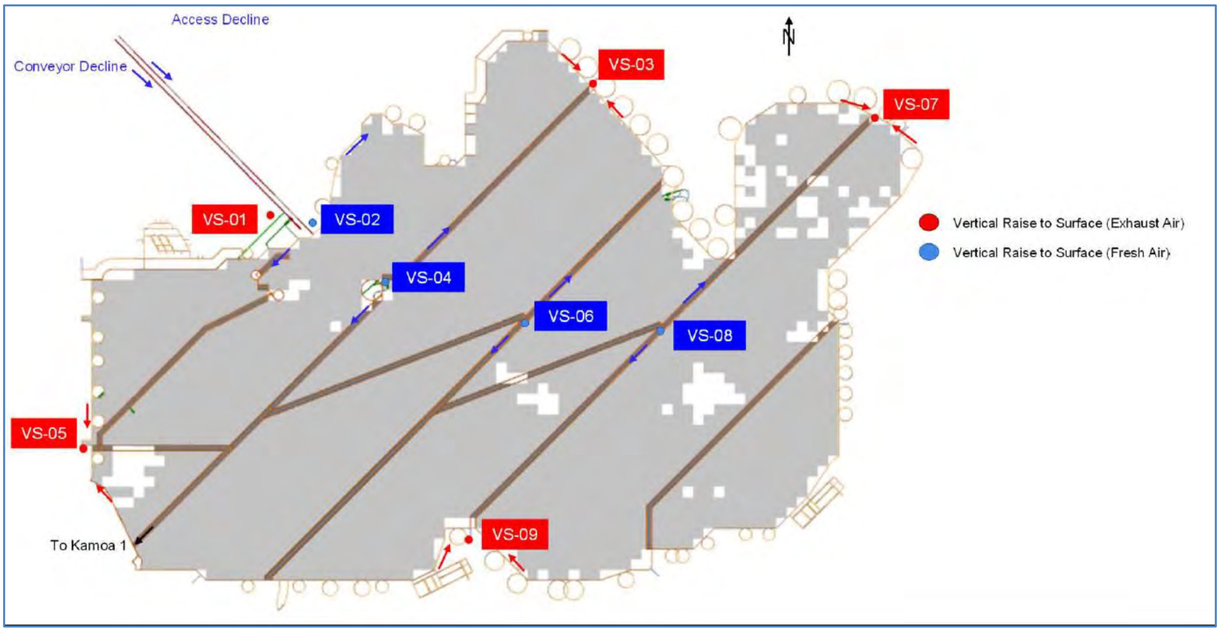
The Kamoa 3 development design and mining zones are shown in Figure 24.10. The ventilation raise and shaft locations are shown in Figure 24.11.

Figure 24.10 Kamoa 3 Mine PEA Development



OreWin, 2023.

Figure 24.11 Kamoa 3 Mine PEA Ventilation Location



OreWin, 2022.

A Kamoa 3 Mineral Resource of approximately 73.1 Mt at 3.14% Cu has been defined with a LOM of 18-years at a production rate of 6.0 Mtpa. Table 24.12 summarises the LOM development and production results.

Table 24.12 Kamoa 3 Mine Life-of-Mine Development and Production Summary

Waste Development	
Lateral (m)	72,737
Lateral (kt)	7,058
Vertical (m)	9,208
Vertical (kt)	566
Production by Mining Method	
Ore Development (m)	1,547
Ore Development (kt)	144
Drift-and-Fill (kt)	72,927
Total Ore Production	
Total Ore Development (kt)	144
Total Production (kt)	72,927
Total Tonnes (kt)	73,070
Diluted Grade	
TCu (%)	3.14
S (%)	1.29
As (%)	0.00
Fe (%)	4.20
AsCu (%)	0.24

1. Notes: Vertical development includes boreholes.
2. Stope shapes designed on an NSR cut-off value of US$100/t NSR.

24.4.2 Kamoa 4

Mine Planning

The preliminary mineable area was obtained using a stope shape optimiser (applied to the Kamoa 4 resource model). Stope optimisation was undertaken on the resource model at a mining cut-off grade of $100/t NSR with the outlier blocks that did not provide a consistent mineable shape removed from the targeted resource. The stope shape optimiser utilised provided assumptions to produce mineable shapes suitable for the chosen mining method.

Similar to Kamoa 3, the primary mining method for the Kamoa 4 deposit is drift-and-fill and was selected based on the results of a trade-off study which compared six different variations of drift-and-fill mining to identify the most suitable. The selected drift-and-fill mining method is explained in detail in Section 16.2.2.

The Kamoa 4 area dimensions are 6.45 km x 4.20 km and the depth of the Kamoa 4 targeted resource is between 31 m–745 m below surface. Access will be via a box-cut and twin declines central to the Kamoa 4 deposit. The drift-and-fill mining zones will be filled and protected by paste fill. The main access to the drift-and-fill production areas is via the perimeter service access drifts and the connection drifts. The extraction ratio of the drift-and-fill zones for all lifts is 98.4%, with the mined tonnes and grades diluted by 1.8% paste fill.

Mine Access

The decline was designed to accommodate two parallel drives with a maximum gradient of ±8.5°. The declines contain a primary mine service access and a conveyor haulage drift. The service decline has dimensions of 6.0 m w x 6.0 m H, with the conveyor decline 7.5 m w x 6.0 m H.

The main conveyor decline is wide enough to accommodate the conveyor as well as large mobile equipment, 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 1,456 m from the portal, 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 shown in Figure 24.12.

From the bottom of the decline, 6 m w x 6.0 m H perimeter service drifts will be driven to the extremities of the deposit. This development will serve as the primary access 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 conveyor system extends to the south-eastern and north-eastern extremities of the orebody and transports material to the main central conveyor decline. The main central decline rock handling system then transports ore to surface via the main central conveyor decline. Conveyor drifts are 7.5 m w x 6.0 m H and are driven at a maximum gradient of ±9.0°.

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

Mine Ventilation

The mine layout schematic illustrating the location of the ventilation shafts is shown in Figure 24.13. Kamoa 4 and Kamoa 5 share conveyor belt infrastructure and service access drifts on the northern side of the deposit, and hence, shared ventilation infrastructure can be utilised. The mine will be supplied with fresh air from four intake ventilation shafts located central to the orebody. The ventilation air will naturally flow through the perimeter drives to the extremities of the orebody. The ventilation will be extracted via five exhaust shafts located at the extremities of the orebody (three at Kamoa 4 and two at Kamoa 5). VS-05 will ventilate the conveyor belt directly to return. 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. The model shows the primary ventilation requirements of 2,500 kg/s at the peak production rate of 6.0 Mtpa and a maximum depth of 745 m.

The mine ventilation cooling requirements were estimated from first principles and confirmed by VentSim models which considered auto-compression, surrounding and broken rock heat, vehicle heat, and all other heat components. The model indicated that no mechanical air cooling will be required for Kamoa 4 as fresh air from surface can ventilate the mine adequately at the required depth of 745 m below surface.

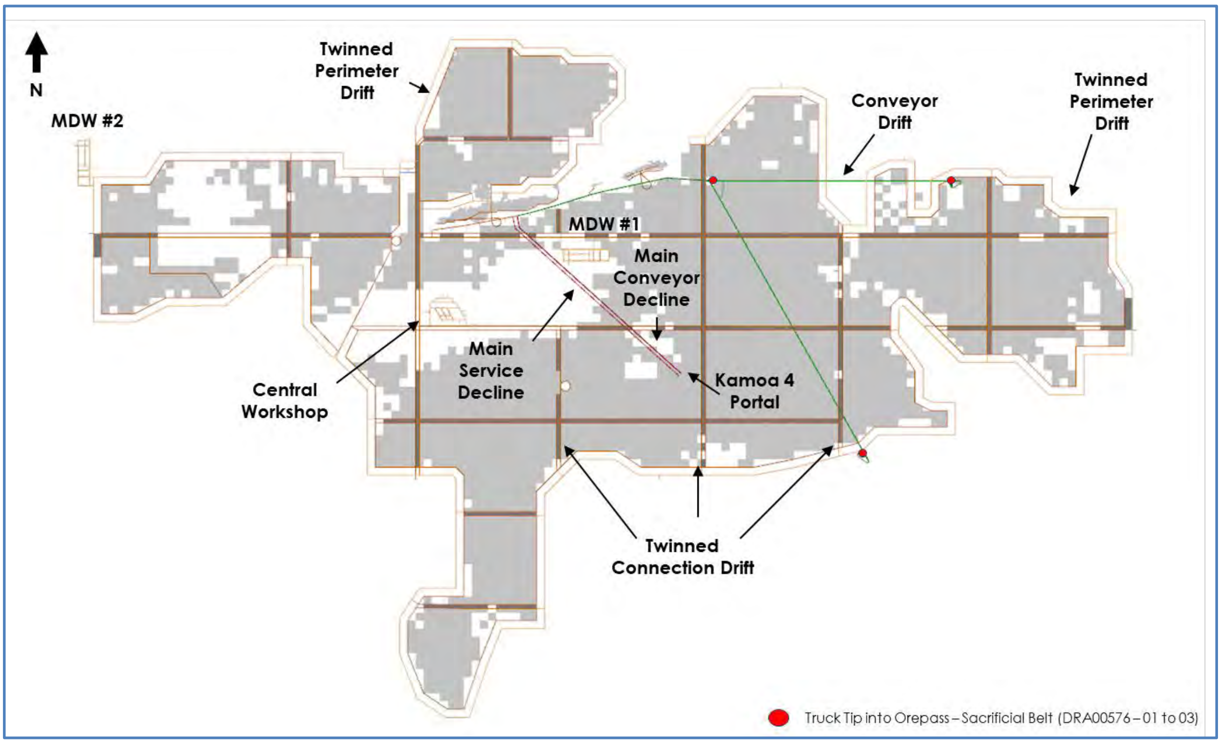
Backfill

Backfill boreholes were strategically designed and located to supply backfill to the drift-and-fill panels over the life of the project.

Mine Development and Production Schedule

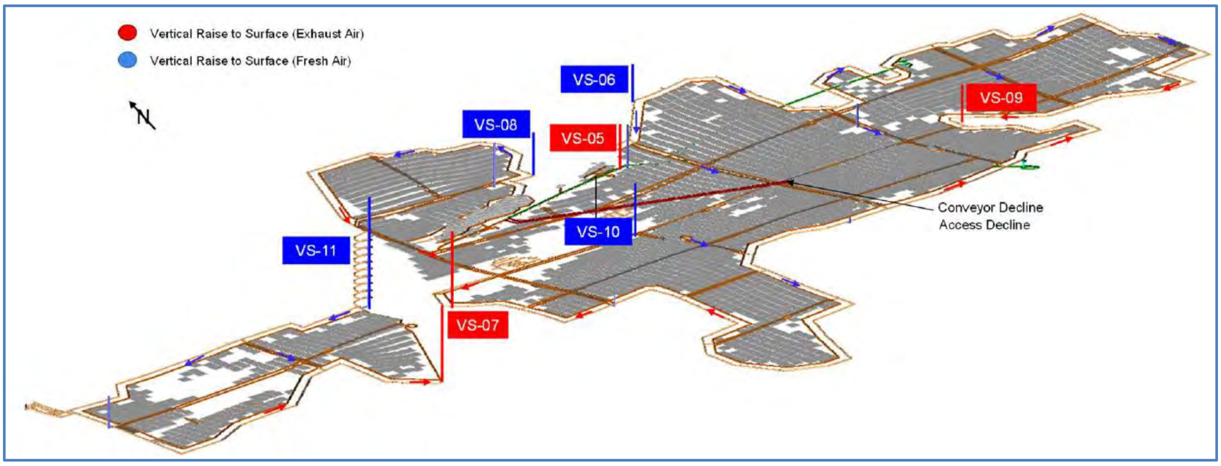
The Kamoa 4 development design and mining zones are shown in Figure 24.12. The ventilation raise and shaft locations are shown in Figure 24.13.

Figure 24.12 Kamoa 4 Mine PEA Development



OreWin, 2023.

Figure 24.13 Kamoa 4 Mine PEA Ventilation Location



OreWin, 2022.

A Kamoa 4 Mineral Resource of approximately 79.1 Mt at 2.95% Cu has been defined with a LOM of 20-years at a production rate of 6.0 Mtpa. Table 24.13 summarises the LOM development and production results.

Table 24.13 Kamoa 4 Mine Life-of-Mine Development and Production Summary

Waste Development	
Lateral (m)	119,128
Lateral (kt)	11,131
Vertical (m)	3,148
Vertical (kt)	185
Production by Mining Method	
Ore Development (m)	1,375
Ore Development (kt)	126
Drift-and-Fill (kt)	78,985
Total Ore Production	
Total Ore Development (kt)	126
Total Production (kt)	78,985
Total Tonnes (kt)	79,111
Diluted Grade	
TCu (%)	2.95
S (%)	1.63
As (%)	0.00
Fe (%)	4.97
AsCu (%)	0.29

1. Notes: Vertical development includes boreholes.
2. Stope shapes designed on an NSR cut-off value of US$100/t NSR.

24.4.3 Kamoa 5

Mine Planning

The preliminary mineable area was obtained using a stope shape optimiser (applied to the Kamoa 5 resource model). Stope optimisation was undertaken on the resource model at a mining cut-off grade of $100/t NSR with the outlier blocks that did not provide a consistent mineable shape removed from the targeted resource. The stope shape optimiser utilised provided assumptions to produce mineable shapes suitable for the chosen mining method.

Similar to Kamoa 4, the primary mining method for the Kamoa 5 deposit is drift-and-fill and was selected based on the results of a trade-off study which compared six different variations of drift-and-fill mining to identify the most suitable. The selected drift-and-fill mining method is explained in detail in Section 16.2.2.

The Kamoa 5 area dimensions are 3.35 km x 3.16 km and the depth of the Kamoa 5 targeted resource is between 100 m–570 m below surface. Access will be via a box-cut and twin declines on the western side of the Kamoa 5 deposit. The Kamoa 5 deposit can also be accessed from the south via Kakula 4. The drift-and-fill mining zones will be filled and protected by paste fill. The main access to the drift-and-fill production areas is via the perimeter service access drifts and the connection drifts. The extraction ratio of the drift-and-fill zones for all lifts is 98.4%, with the mined tonnes and grades diluted by 1.8% paste fill.

Mine Access

The decline was designed to accommodate two parallel drives with a maximum gradient of ±9°. The declines contain twin mine service access drifts. Ore material will utilise the Kamoa 5 southern conveyor system to transport material to surface via the Kamoa 4 conveyor network. Waste material can be transported to surface using the western service access declines or via the Kamoa 4 access in the south. The service declines have dimensions of 6.0 m w x 6.0 m H.

The western declines utilise one way traffic flow in order to minimise load and haul congestion. Approximately 802 m from the western portal, access drifts are developed off the main service decline to the perimeter declines. These drifts also provide access to the main workshop and truck tip area, as shown in Figure 24.14.

From the bottom of the decline, 6 m w x 6.0 m H perimeter service drifts will be driven to the extremities of the deposit. This development will serve as the primary access 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 conveyor system extends from the southern extremity of the orebody and transports material to the Kamoa 4 conveyor network. The Kamoa 4 rock handling system then transports ore to surface via the Kamoa 4 main central conveyor decline. Conveyor drifts are 7.5 m w x 6.0 m H and are driven at a maximum gradient of ±9.0°.

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

Mine Ventilation

The mine layout schematic illustrating the location of the ventilation shafts is shown in Figure 24.15. Kamoa 5 and Kamoa 4 share conveyor belt infrastructure and service access drifts on the southern side of the deposit, and hence, shared ventilation infrastructure can be utilised. The mine will be supplied with fresh air from two intake ventilation shafts located to the north of the orebody. The ventilation air will naturally flow through the perimeter drives to the extremities of the orebody. The ventilation will be extracted via five exhaust shafts located at the extremities of the orebody (three at Kamoa 4 and two at Kamoa 5). VS-05 (Kamoa 4) will ventilate the conveyor belt directly to return. 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. The model shows the primary ventilation requirements of 2,870 kg/s at the peak production rate of 3.0 Mtpa and a maximum depth of 570 m.

The mine ventilation cooling requirements were estimated from first principles and confirmed by VentSim models which considered auto-compression, surrounding and broken rock heat, vehicle heat, and all other heat components. The model indicated that no mechanical air cooling will be required for Kamoa 5 as fresh air from surface can ventilate the mine adequately at the required depth of 570 m below surface.

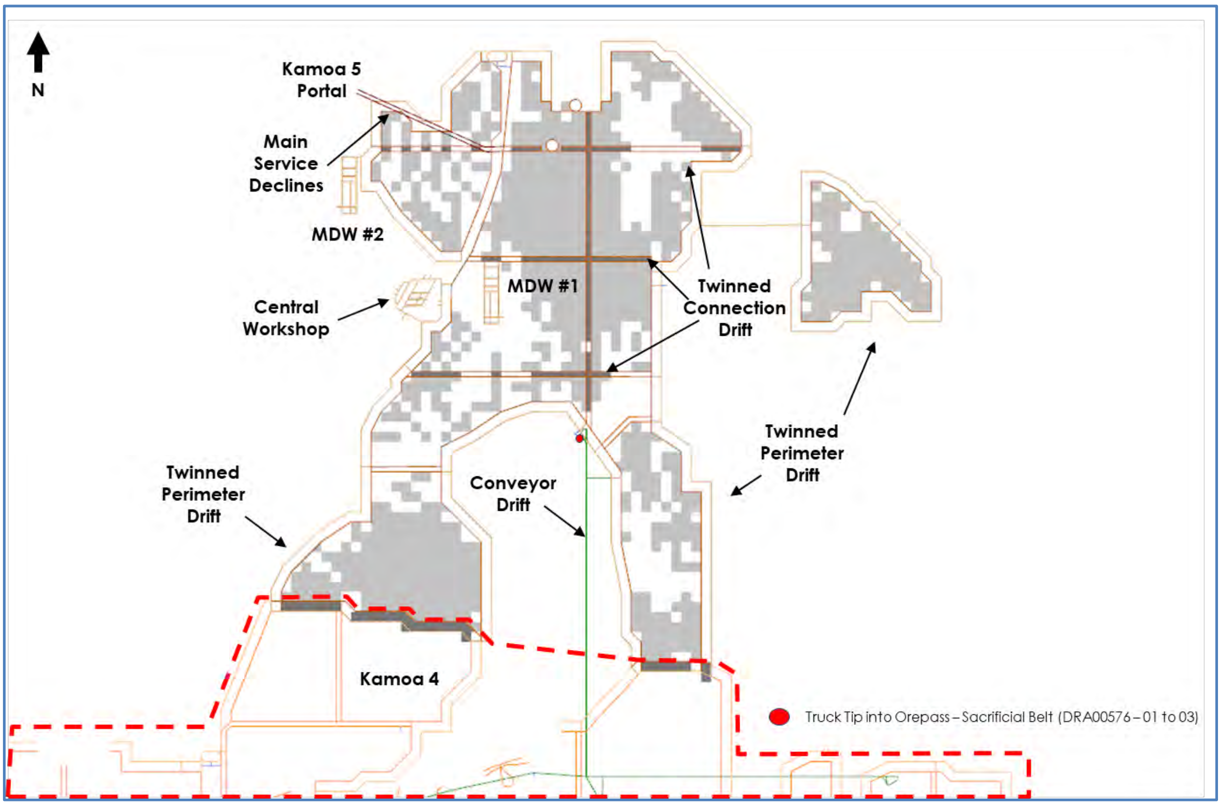
Backfill

Backfill boreholes were strategically designed and located to supply backfill to the drift-and-fill panels over the life of the project.

Mine Development and Production Schedule

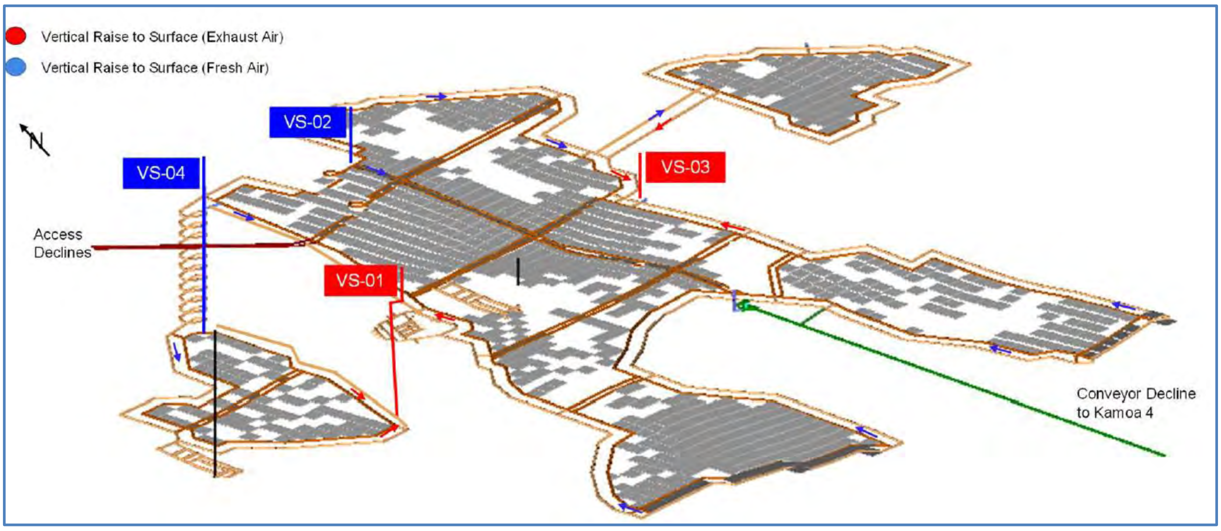
The Kamoa 5 development design and mining zones are shown in Figure 24.14. The ventilation raise and shaft locations are shown in Figure 24.15.

Figure 24.14 Kamoa 5 Mine PEA Development



OreWin, 2023.

Figure 24.15 Kamoa 5 Mine PEA Ventilation Location



OreWin, 2022.

A Kamoa 5 Mineral Resource of approximately 19.3 Mt at 3.04% Cu has been defined with a LOM of nine-years at a production rate of 3.0 Mtpa. Table 24.14 summarises the LOM development and production results.

Table 24.14 Kamoa 5 Mine Life-of-Mine Development and Production Summary

Waste Development	
Lateral (m)	59,169
Lateral (kt)	5,602
Vertical (m)	1,638
Vertical (kt)	82
Production by Mining Method	
Ore Development (m)	867
Ore Development (kt)	80
Drift-and-Fill (kt)	19,227
Total Ore Production	
Total Ore Development (kt)	80
Total Production (kt)	19,227
Total Tonnes (kt)	19,307
Diluted Grade	80
Tcu (%)	3.04
S (%)	2.42
As (%)	0.00
Fe (%)	5.80
AsCu (%)	0.32

1. Notes: Vertical development includes boreholes.
2. Stope shapes designed on an NSR cut-off value of US$100/t NSR.

24.4.4 Kamoa 6

Mine Planning

The preliminary mineable area was obtained using a stope shape optimiser (applied to the Kamoa 6 resource model). Stope optimisation was undertaken on the resource model at a mining cut-off grade of $100/t NSR with the outlier blocks that did not provide a consistent mineable shape removed from the targeted resource. The stope shape optimiser utilised provided assumptions to produce mineable shapes suitable for the chosen mining method.

Similar to Kamoa 5, the primary mining method for the Kamoa 6 deposit is drift-and-fill and was selected based on the results of a trade-off study which compared six different variations of drift-and-fill mining to identify the most suitable. The selected drift-and-fill mining method is explained in detail in Section 16.2.2.

The Kamoa 6 area dimensions are 2.48 km x 1.10 km and the depth of the Kamoa 6 targeted resource is between 193 m– 250 m below surface. Access will be via a box-cut and twin declines south of the Kamoa 6 deposit. The drift-and-fill mining zones will be filled and protected by paste fill. The main access to the drift-and-fill production areas is via the perimeter service access drifts and the connection drifts. The extraction ratio of the drift-and-fill zones for all lifts is 98.4%, with the mined tonnes and grades diluted by 1.8% paste fill.

Mine Access

The decline was designed to accommodate two parallel drives with a maximum gradient of ±8.5°. The declines contain twin mine service access drifts. The service declines have dimensions of 6.0 m W x 6.0 m H.

The southern declines utilise one way traffic flow in order to minimise load and haul congestion. Approximately 1,584 m from the portal, access drifts are developed off the main service decline to the perimeter declines as shown in Figure 24.16.

From the bottom of the decline, 6 m W x 6.0 m H perimeter service drifts will be driven to the extremities of the deposit. This development will serve as the primary access 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.

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

Kamoa 6 does not feature a conveyor system and hence, all material must be trucked to surface via the southern access declines. The southern declines utilise one way traffic flow to minimise load and haul congestion.

Mine Ventilation

The mine layout schematic illustrating the location of the ventilation shafts is shown in Figure 24.17. The mine will be supplied with fresh air from the declines and one intake ventilation shaft located to the north of the orebody. The ventilation air will naturally flow through the perimeter drives to the extremities of the orebody. The ventilation will be extracted via an exhaust shaft located central to the orebody. 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. The model shows the primary ventilation requirements of 500 kg/s at the peak production rate of 1.0 Mtpa and a maximum depth of 250 m.

The mine ventilation cooling requirements were estimated from first principles and confirmed by VentSim models which considered auto-compression, surrounding and broken rock heat, vehicle heat, and all other heat components. The model indicated that no mechanical air cooling will be required for Kamoa 6 as fresh air from surface can ventilate the mine adequately at the required depth of 250 m below surface.

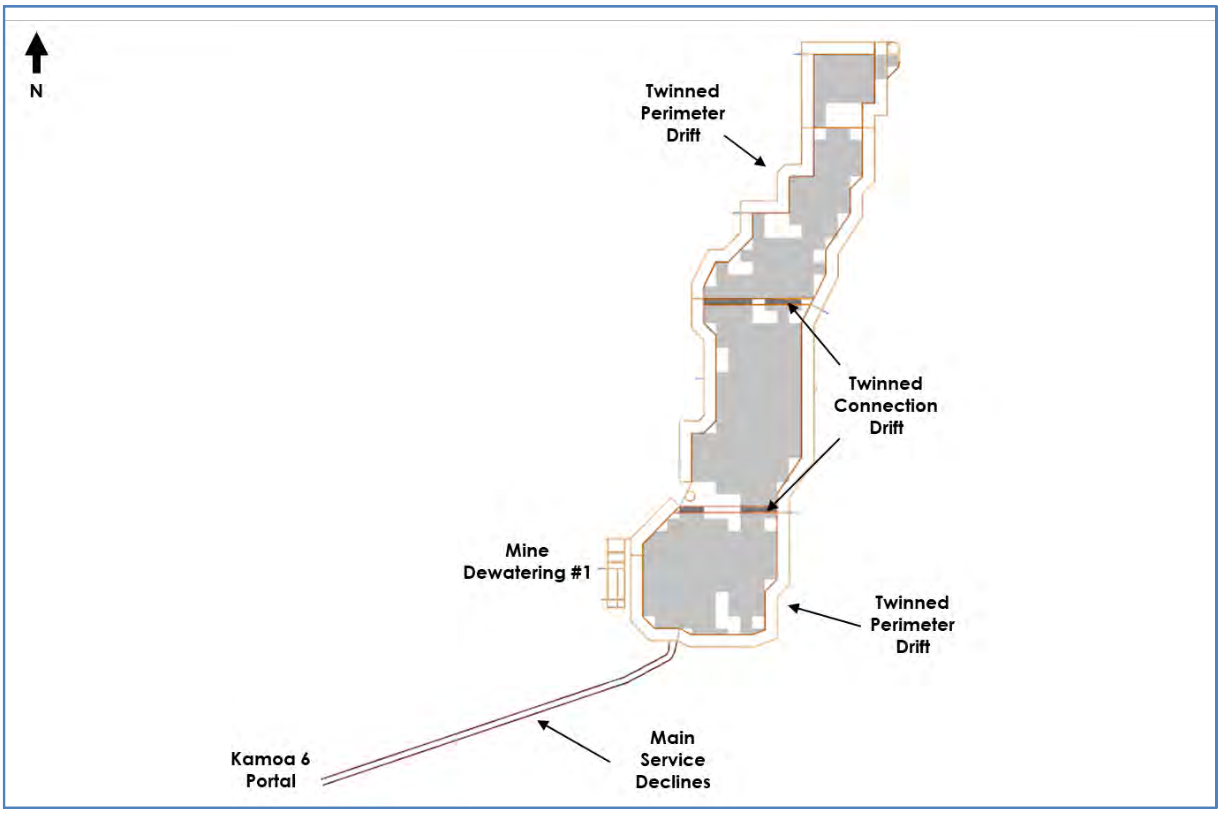
Backfill

Backfill boreholes were strategically designed and located to supply backfill to the drift-and-fill panels over the life of the project.

Mine Development and Production Schedule

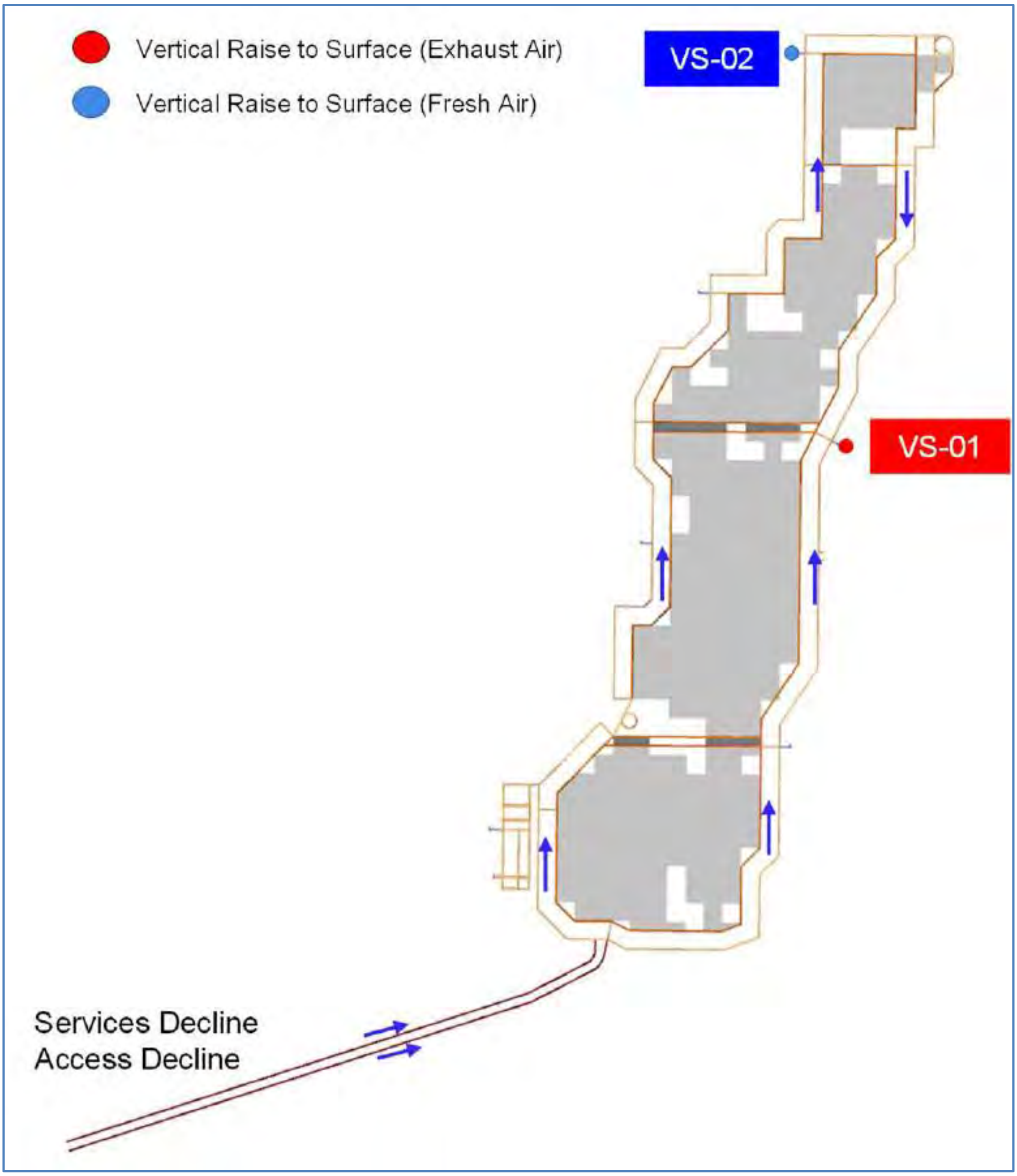
The Kamoa 6 development design and mining zones are shown in Figure 24.16. The ventilation raise and shaft locations are shown in Figure 24.17.

Figure 24.16 Kamoa 6 Mine PEA Development



OreWin, 2023.

Figure 24.17 Kamoa 6 Mine PEA Ventilation Location



OreWin, 2022.

A Kamoa 6 Mineral Resource of approximately 9.75 Mt at 3.42% Cu has been defined with a LOM of 13-years at a production rate of 1.0 Mtpa. Table 24.15 summarises the LOM development and production results.

Table 24.15 Kamoa 6 Mine Life-of-Mine Development and Production Summary

Waste Development	
Lateral (m)	20,037
Lateral (kt)	1,885
Vertical (m)	1,787
Vertical (kt)	118
Production by Mining Method	
Ore Development (m)	150
Ore Development (kt)	13
Drift-and-Fill (kt)	9,732
Total Ore Production	
Total Ore Development (kt)	13
Total Production (kt)	9,732
Total Tonnes (kt)	9,745
Diluted Grade	
Tcu (%)	3.42
S (%)	2.36
As (%)	0.00
Fe (%)	6.03
AsCu (%)	0.64

1. Notes: Vertical development includes boreholes.
2. Stope shapes designed on an NSR cut-off value of US$100/t NSR.

24.5 Kamoa-Kakula 2023 PEA Processing and Infrastructure

The Kamoa-Kakula 2023 PEA assesses a nine-year mine life extension of the Kamoa-Kakula Copper Complex, in addition to the Kamoa-Kakula 2023 PFS. This case includes the addition of four new underground mines in the Kamoa area (called Kamoa 3, 4, 5, and 6) to maintain the overall production rate of up to 19.2 Mtpa.

As the resources at the Kakula, Kakula West, Kamoa 1, Kansoko Sud, and Kamoa 2 mines are mined out, production would begin sequentially at the four other mines in the Kamoa complex to maintain throughput of 19.2 Mtpa to the then existing concentrator and smelter complex.

Each mining operation is expected to be a separate underground mine with a shared processing facilities and surface infrastructure located at Kakula and Kamoa. Included in this scenario is the construction of an overland conveyor over the years 2045 and 2046, capable of conveying required material from the Kamoa mines to the Kakula processing complex.

The annual production results including processing, concentrate and smelter production are shown in Table 24.9 to Table 24.11.

The 19.2 Mtpa scenario is comprised of four concentrator plants spread between the Kakula and Kamoa. These concentrator plants are configured in a combination of Kakula and Kamoa process flows.

Processing Summary:

- Total processing capacity of 19.2 Mtpa.
- Kakula Complex (2 x 4.6 Mtpa).
- Kamoa Complex (2 x 5.0 Mtpa).

Overland Conveyor from Kamoa mines will be used to convey required ore to the Kakula complex. The ore conveying commences in year 2047 when production from Kakula and Kakula West mines gradually ramps down from 9.2 Mtpa, and production from Kamoa 3 – 6 mines sequentially start ramping up. Kamoa 2 – 6 mines will sustain 19.2 Mtpa production rate from year 2049 onwards, out of which 10.0 Mtpa will be processed at the Kamoa processing complex and the rest will be conveyed to the Kakula processing complex via overland conveyor.

24.5.1 Kamoa and Kakula Processing Complex

The throughput from Kakula Phase 1 and 2 concentrators is increased to 9.2 Mtpa by the imminent completion of the debottlenecking programme. The Kakula complex includes a ROM stockpile to feed a 9.2 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 products, i.e. a high-grade, and a medium-grade product. These two concentrate products are combined to form the final concentrate. Refer to Section 24 for more detail on the Kakula complex concentrator design, production capacity and concentrator process.

Kamoa processing complex includes two 5.0 Mtpa concentrators. Construction of the first 5.0 Mtpa concentrator is ongoing, which is targeted to be completed in Q4’24. An another adjacent 5.0 Mtpa concentrator will be added to Kamoa processing complex later in the decade to take the total processing capacity of the Kamoa processing complex to 10.0 Mtpa, bringing overall production up to 19.2 Mtpa. Refer to Section 24for more detail on the Kamoa complex concentrator design, production capacity and concentrator process.

Concentrators at Kakula and Kamoa processing complexes will be sufficient to process the material mined from PEA mines (Kamoa 3 – 6) throughout the LOM. Once the Kakula and Kakula West material starts to deplete, the Kamoa 3, Kamoa 4, Kamoa 5 and Kamoa 6 Mines are brought online sequentially to maintain the 19.2 Mtpa processing rate. The material mined from Kamoa mines exceeding the processing capacity of Kamoa processing complex will be conveyed to the Kakula processing complex via Overland Conveyor. The material from Kamoa ROM stockpile will be extracted by a duty / standby apron feeder arrangement, prior to discharge onto Overland Conveyor.

24.5.2 Kamoa-Kakula 2023 PEA Smelter

The smelter is designed to process 1,200 ktpa concentrate, producing up to 500 ktpa blister copper and an average 725 ktpa (ranging from 650 to 800 ktpa) of high strength sulfuric acid. The concentrate produced from the material mined from PEA mines (Kamoa 3 – 6) will be processed using the existing smelter for the LOM. Refer to Section 17 for more detail on the smelter design, production capacity, schedule and smelting process.

24.5.3 Kamoa-Kakula 2023 PEA Infrastructure

The infrastructure for the Kamoa-Kakula 19.2 Mtpa PEA must support two 4.6 Mtpa and two 5.0 Mtpa concentrator circuits, at Kakula and Kamoa processing complexes, 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 wastewater, buildings, accommodations, security, and medical services.

24.5.3.1 Power

Power for the Kamoa-Kakula Project is to be sourced from the DRC's state owned power company (SNEL, Société Nationale d'Electricité) electrical interconnected grid. The power supply available on-site will be sufficient to sustain 19.2 Mtpa production including the PEA mines. Refer to Section 18 for more detail on the power requirements and infrastructure.

24.5.3.2 TSF

The Kamoa-Kakula 19.2 Mtpa PEA considered a new TSF at the Kamoa processing complex to allow for the additional tailings tonnage. The new TSF will come online by end of year 2048.

24.5.3.3 Site Access and Transport

The main access road to the Kakula and Kamoa are constructed and being used to allow for future mining activities required for the Kamoa-Kakula Copper Project. This road gives access from Kolwezi to Kakula and Kansoko Sud/Kamoa 1 mine and is divided into two sections:

- Section 1: Section from Kolwezi Mines turnoff to Kansoko Sud/Kamoa 1.
- Section 2: Kansoko Sud/Kamoa 1 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.

24.5.3.4 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.

24.5.3.5 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.

24.5.3.6 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.

24.5.3.7 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.

24.5.4 Kamoa-Kakula 2023 PEA Process and Infrastructure Costs

The Kamoa-Kakula 2023 PEA will continue sustaining 19.2 Mtpa once the PFS mines deplete. The following inputs and documents were identified and used in compiling the capital cost estimate:

Estimate Combined Rev A - 20230124.xlsxCosts 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 fill 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.

24.5.5 Comments on Section 24.5

ROM ore is assumed to have a top size 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 top size of 550 mm.

The plant design is based on a 53 µm flotation feed P80 and a 10 µm regrind P80 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.

During the infrastructure planning for the Kamoa-Kakula 2023 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.

24.6 Kamoa-Kakula 2023 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 Kamoa-Kakula 2023 PFS. Allowances for operating the Kakula and Kamoa processing complexes 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, 2019), and conform to the requirements of CIM Definition Standards (2014). MSA has checked the data used to construct the resource model, the methodology used to construct them (Datamine macros) and has validated the resource model. MSA finds the Kamoa and Kakula resource models at Indicated or Measured classification to be suitable to support at least pre-feasibility 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, although this has not yet been the case in mining operations.
  - Delineation drill programmes 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.
  - Assumptions used to generate the data for consideration of reasonable prospects of eventual economic extraction for the Kakula deposit.
- Metallurgical Recovery Assumptions at Kamoa.
  - Variability testwork has been conducted on portions of the Kamoa deposit and therefore the average recoveries used in the cut-off grade assessment may differ from actual performance. Areas of supergene mineralisation are likely to require different metallurgical parameters, however these areas make up only a small part of the deposit.
- Commodity prices and exchange rates.
- Cut-off grades.

25.2 Kamoa-Kakula IDP23

The Kamoa-Kakula IDP23 includes an update of the Kakula, and Kamoa Mineral Reserve, and updates of the preliminary economic assessment (PEA), including analysis of the Kamoa 3, Kamoa 4, Kamoa 5, and Kamoa 6 Mineral Resource.

The Kamoa-Kakula 2023 PFS has identified a Mineral Reserve and development path that has confirmed significant value in the Kamoa, and Kakula deposits.

The Kamoa-Kakula 2023 PFS indicates the combined Kakula, Kakula West, Kansoko Sud, Kamoa 1, and Kamoa 2 mine plans using the 9.2 Mtpa Kakula processing complex, and 10.0 Mtpa Kamoa processing complex, including on-site smelting, generates significant value.

The analysis in the Kamoa-Kakula 2023 PEA assesses a nine-year mine life extension of the Kamoa-Kakula Copper Complex, in addition to the Kamoa-Kakula 2023 PFS. This case includes the addition of four new underground mines in the Kamoa area (called Kamoa 3, 4, 5, and 6) to maintain the overall production rate of up to 19.2 Mtpa.

25.3 Mineral Reserve Estimate

25.3.1 Kakula Mineral Reserve Estimate

Mineral Reserves for the Kakula 2023 PFS conform to the requirements of CIM Definition Standards (2014). The utilised development processes and cost estimates to the level of accuracy required to state reserves and support a pre-feasibility level study. Areas of uncertainty that may impact the Mineral Reserve Estimate include:

- Commodity prices and exchange rates.
- The continuity and dip of the ore will need to be better defined prior to, and during the mining stages.
- The amount of groundwater present in the orebody during the mining cycle.
- 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 PFS 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 2023 PFS conform to the requirements of CIM Definition Standards (2014). The development processes and cost estimates to the level of accuracy required to state reserves and support a pre-feasibility level study. Areas of uncertainty that may impact the Mineral Reserve Estimate include:

- Commodity prices and exchange rates.
- The continuity and dip of the ore will need to be better defined prior to and during the mining stages.
- The areas previously identified for the Kamoa-Kakula 2023 PFS 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 pilot plant conducted on the Kakula material as part of the Kamoa-Kakula 2023 PFS.

It is the opinion of the qualified person that the Smelter design described in the report is acceptable. It is not supported by metallurgical testwork programmes but draws on industry experience of many similar copper flash smelting, converting, slag cleaning, and sulfuric acid production installations, extending back worldwide at least 70-years. It also utilises fundamental thermodynamic relationships to estimate the mass and energy balances for the Smelter. Calculations, design criteria and conclusions were included in the respective PFS smelter designs and appeared to be consistent with the extensive body of published data for similar smelters.

25.5 Infrastructure

Surface infrastructure construction contractors are currently established to commence with Phase 3, or in the process of site establishment. The general working environment is well understood after the completion of Phase 1 and 2, with constraints of services, and access to site already mitigated. This serves as high confidence in estimating rates and construction durations used across the Kamoa-Kakula 2023 PFS.

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. Additional power will be required to meet the peak demands of Phase 4 operations. It was assumed that an additional turbine at Inga power plant will be refurbished. Although the cost and scope is well understood, the actual agreement of doing that is not yet finalised by the time this study was concluded.

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 PFS.

26 RECOMMENDATIONS

26.1 Further Assessment

The Kamoa-Kakula 2023 PFS described development scenarios for Kamoa, and Kakula, deposits that expand on the initial two phases of the project. The first, Phase 3 is already significantly advanced and is to be followed by Phase 4 for a total processing capacity of 19.2 Mtpa. A holistic approach should be undertaken to optimise the project value. The two additional phases identified in the Kamoa-Kakula 2023 PFS provide the current development plan for the Kamoa-Kakula Mineral Resources. As development continues each stage of the project should be analysed and redefined.

The key areas for further studies are:

- The Kamoa-Kakula 2023 PFS has identified a significant Mineral Reserve to be mined. Performance at the operation has successfully achieved production rates above those planned in the previous studies. The next study work should consider the timing and implementation of the Phase 4 project which is earmarked to be brought on-line once the Phase 3 project reaches steady state. The next study work should consider optimisation for increased production at rates above 20 Mtpa from the combined Kakula and Kamoa mines.
- At Kamoa the groundwork to support bringing forward the timing of Phase 4 should be conducted. This should include further exploration drilling and upgrading of the Mineral Resources and targeted technical studies to support the Phase 4 project implementation.
- The Kamoa-Kakula 2023 PEA indicates that there is potential value in extending the life of the Kamoa-Kakula operation with the addition of Kamoa 3, Kamoa 4, Kamoa 5, and Kamoa 6 mines. To identify this potential, further study will be needed. These studies will be undertaken using a holistic approach into the long-term options to maximise the efficient extraction of the Kamoa-Kakula Mineral Resources.

Other areas of further study include:

- Revisions and updates of the long-term holistic mine planning as the Mineral Resources Mineral Reserves 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.
- 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.
- Progress technical studies to better understand and gain knowledge of technical aspects to support the successful implementation of the studies.
- Continue the underground exploration programme at Kakula to further define the orebody and assist with the location of the perimeter and connection drifts.
- Further hydrological studies and data evaluation to better determine impacts on underground mining conditions and productivities.

- 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.

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 plans at Kakula and Kamoa is to continue infill drilling in support of the current mine development, and to define the edges of the higher grade material.

26.3 Underground Mining

The following is a list of mining recommendations for the Project:

- 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, geological structures 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.
- Further 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 and thermal conductivities of the ore bodies especially at depth to allow for ventilation and refrigeration modelling.
- Develop an operating philosophy to optimise waste rock going into room-and-pillar goaf, and drift-and-fill areas.
- Perform a detailed simulation of the underground traffic flow at peak production and adjust the mine rock handling systems to optimize material handling and costs.
- 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.
- Investigate the possibility of adding sand to the tailings at Kakula to reduce the binder quantity.
- Complete a Feasibility Study on the Kamoa-Kakula deposits.

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.

Some assumptions within the ongoing studies are based upon information which was available at the time. More detailed analysis around the geological structures and geotechnical conditions specific to the Kakula West and Kamoa 1 mines should be conducted in support of the planned mine production.

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.
- 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 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. 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 Smelter

Solar power with battery storage may be an alternative to emergency diesel generators that is worth considering.

Use of burnt lime flux in the smelter, instead of limestone, will reduce power consumption. However, the hazards to operation and maintenance personnel of using burnt lime, and the potential consequences of the marked exothermic reaction if the material is wettened, must be considered carefully in the detailed design, and operating practices for to be developed for the smelter.

Vendor project references for the combination of diesel burners and supplementary pulverised coal in a controlled flash smelting or DBF operation will provide more confidence if they can be obtained.

Granulation of SCF slag before re-treatment by flotation could be considered as an alternative to the proposed slow cooling and crushing arrangement, which requires several hundred slag ladles. Comparative comminution and flotation testwork should be carried out to support the final design solution.

The mass balance as well as the number of equipment items in slag flotation, and product handling, should be investigated further to ensure that product grade objectives, recovery targets and plant availability will all be achieved.

26.6 Infrastructure

It is recommended that further studies of the Kamoa-Kakula infrastructure should be undertaken. The assumption that the refurbishment of one of the INGA power plant turbines will be necessary to meet the project's power requirements carries some inherent risk. One of the main risks is the unavailability of the turbine for refurbishment, which could result in delays in the ramp-up of production. The requirement for additional power is expected to become necessary from 2027, and it is crucial for the project to have a reliable source of power to meet its demand.

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