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Technical Report

Kamoa-Kakula Mineral Reserve and Mineral Resource Update Technical Report

Kamoa Copper SA

Democratic Republic of Congo (DRC)

In accordance with the requirements of National Instrument 43-101 “Standards of Disclosure for Mineral Projects” of the Canadian Securities Administrators

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Effective date 31 March 2026

1 Summary

1.1 Introduction

Ivanhoe Mines Ltd. (Ivanhoe) is a mineral exploration, development, and mining company, whose principal properties are located in Africa. The Ivanhoe strategy is to build a global, commodity-diversified mining and exploration company. Ivanhoe has focused on exploration within the Central African Copperbelt, and the Bushveld Complex.

Ivanhoe currently has three key assets: An interest in the Kamoa-Kakula Copper Complex; 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 in the Central African Copper belt adjoining the Kamoa-Kakula Project, known as the Western Foreland.

This Technical Report addresses an update of the Mineral Resources and Mineral Reserves of the Kamoa-Kakula complex in the Democratic Republic of the Congo (DRC).

1.2 Effective dates

- The effective date of this Report: 31 March 2026.
- 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 and Kakula Mineral Resources were depleted to account for annual production and have an effective date of 31 December 2025.
- Date of the Mineral Reserve estimate for Kamoa-Kakula: 31 December 2025.
- Date of the supply of legal information supporting mineral tenure: 24 March 2026.
- All cost and cashflow estimates are based on Real, Quarter 1 2026 US Dollar money terms.

1.3 Property description and location

The Kamoa-Kakula Project is situated in the Mutshatsha territory, Lualaba Province, Democratic Republic of Congo (DRC), approximately 25 kilometres (km) west of the town of Kolwezi and 270 km west of the regional centre of Lubumbashi. The Project is centered at approximately latitude 10°46'S and longitude 25°15'E.

The Project is held by Kamoa Copper SA (KCSA) which is owned 80% by Kamoa Holding Limited and 20% directly by the DRC government. Kamoa Holding is owned 49.5% by Ivanhoe Mines Ltd, 49.5% by Zijin Mining Group Co. Ltd, and 1% by Crystal River Global Limited, resulting in effective interests in the Project of approximately 39.6% for each of Ivanhoe and Zijin, 0.8% for Crystal River, and 20% for the DRC. The DRC's 20% interest comprises a 5% non-dilutable interest transferred pursuant to the DRC Mining Code at the time of the shareholders' general meeting in September 2012, and a further 15% interest transferred by share transfer agreement in November 2016.

The Project comprises three exploitation permits — PE No. 12873, 13025, and 13026 — covering a combined area of approximately 39,316 hectares, all granted on 20 August 2012 with an expiry date of 19 August 2042. The permits are valid for 25 years and are renewable for periods not exceeding 15 years until end of mine life, subject to satisfaction of the conditions set out in the DRC Mining Code. The permits cover a range of metals including copper, cobalt, silver, gold, nickel, zinc, and associated minerals.

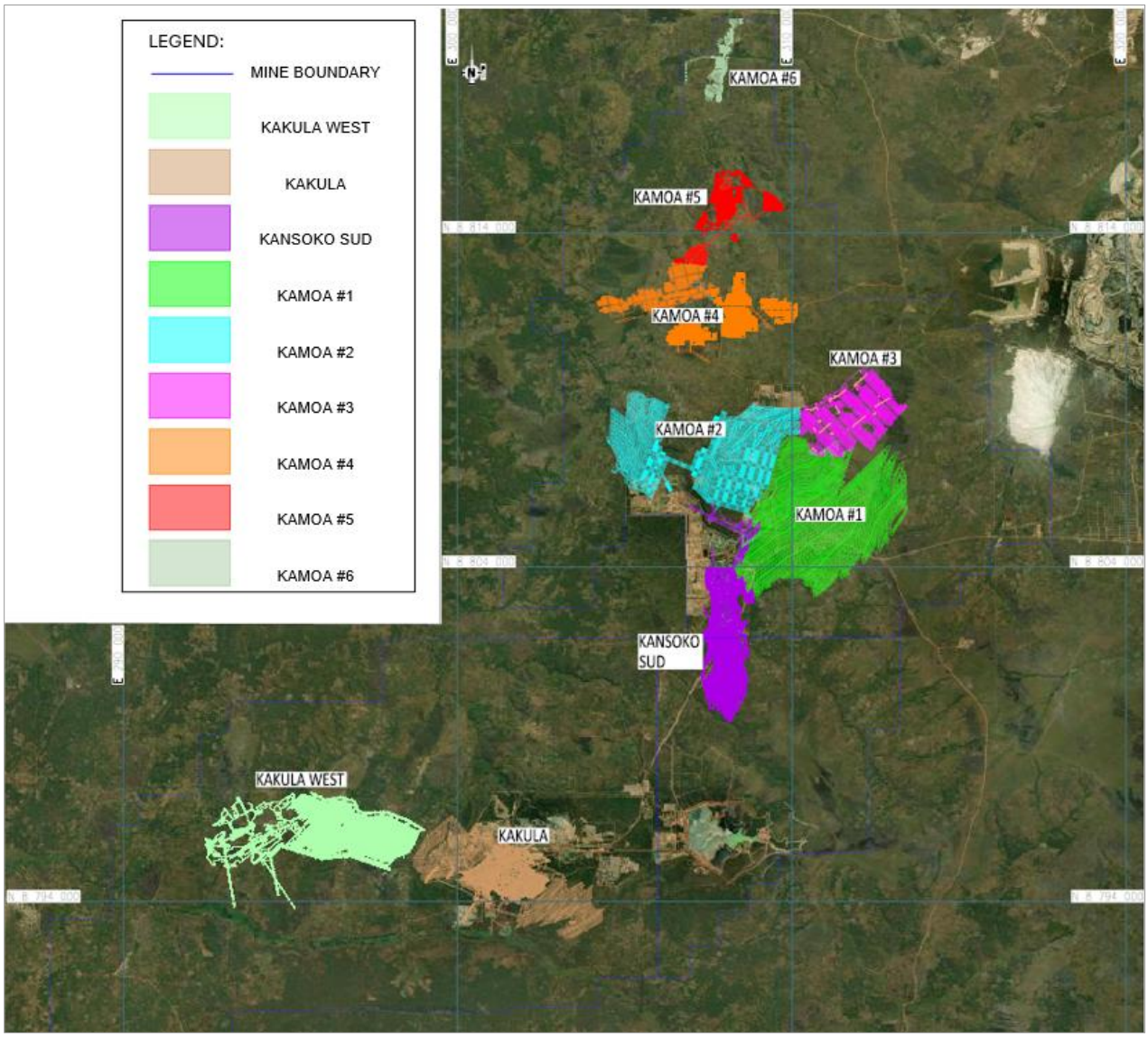
All mineral deposits in the DRC are state-owned; however, the holder of an exploitation permit acquires ownership of products for sale. Mining activities are governed by the DRC Mining Code (Law No. 007/2002, as amended by Law No. 18/001 dated March 2018) and the Mining Regulations. Mining royalties are payable at a rate of 3.5% of gross commercial value for non-ferrous and base metals, including copper.

Environmental obligations are governed by the Mining Code and require an approved Environmental and Social Impact Study (ESIS) and Environmental and Social Management Plan (ESMP). KCSA submitted an updated ESIS in April 2022, for which the environmental certificate was issued on 13 July 2022. Environmental audits are required every two years from approval of the initial ESIS. A further updated ESIS submission is in progress.

KCSA holds no formal surface rights title over the Project area; however, it has been authorized by the Governor of Lualaba Province to occupy land required for its mining activities. Compensation and resettlement programs have been completed or are in progress for affected households within the mine footprints in accordance with DRC law. There are no property agreements in place that are relevant to this Technical Report.

The Qualified Persons (QPs) are not aware of any environmental liabilities, legal disputes, or impediments to the Project that would materially affect the Mineral Reserve estimate.

Figure 1.1 Kamoa-Kakula site plan



Source: DRA, 2026.

1.4 Accessibility, climate, local resources, infrastructure, and physiography

1.4.1 Accessibility

The Project is accessible by air via the international airports at Lubumbashi (290 km east) and Ndola, Zambia (200 km south-east of Lubumbashi), with regular connecting flights to the regional centre of Kolwezi approximately 25 km to the east of the Project, at a flying time of approximately 45 minutes from Lubumbashi. Road access to the Project from Kolwezi is via a purpose-built gravel road joining the Kolwezi-Lubumbashi tarred road at the Kolwezi airport. Sealed on-site roads connect the Kamoa Camp, Kamoa mines, Kansoko Mine, Kakula Mine and Kakula processing and smelter Complex. Rail infrastructure in the region connects via the north–south corridor through Zambia and Zimbabwe to South African ports, and via the Lobito Corridor westward through Angola to the Atlantic port of Lobito — the DRC section of this western line remains in poor condition, though a 30-year concession for the Angolan section was awarded in November 2022 to a consortium including Trafigura, Mota-Engil Africa, and Vecturis SA.

1.4.2 Climate

The Project experiences a tropical climate with a wet season from October to March (peak precipitation December to February) and a dry season from April to September. Annual rainfall averages approximately 1,225 mm. Average annual temperatures range from 16°C to 28°C, with a mean of approximately 20.6°C. Prevailing winds are from the east–south-east and south-east.

1.4.3 Workforce and infrastructure

The workforce is drawn from local villages, Kolwezi, Lubumbashi, other regions of the DRC, and internationally, with minimized and regulated expatriate participation. Transport is provided by KCSA from Kolwezi and surrounding areas. Existing site infrastructure supports current underground mining and processing operations at the Kakula, Kansoko Sud, and Kamoa 1 and 2 mines, and the Phase 1, 2, and 3 concentrator facilities.

1.4.4 Power

The bulk power supply is sourced from La Société Nationale d'Électricité (SNEL), the national power utility of the 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 extended the loan agreement with SNEL, for the upgrade of unit 5 (G25) at Inga II HPP in the South-west DRC. The upgrade of unit 5 (G25) was completed in November 2025 and the available capacity will increase to 125 MW in May 2027.

As of the effective date of this report, KCSA is busy with the installation of the following Solar PV and BESS systems by IPPs. The Solar / BESS plants are baseload specified and can therefore also serve as standby supply.

- Kamoa #1 – 30 MW baseload due for commissioning June 2026.
- Kamoa #2 - 30 MW baseload due for commissioning August 2026.
- The following systems are in the planning phase and will be installed at Kakula.
- Kakula #1 – 30 MW baseload planned for commissioning December 2026.
- Kakula #2 – 30 MW baseload planned for commissioning January 2028.

The following Solar / BESS systems are planned to supply power into the future Kamoa KCS #2 substation.

- Kamoa #3 – 30 MW baseload planned for commissioning December 2029.
- Kamoa #4 – 30 MW baseload planned for commissioning January 2029.

1.4.5 Physiography

The Project is located on a north–north-east to south–south-west trending ridge at elevations between 1,300 metres (m) and 1,540 m above mean sea level, dissected by the Mukanga, Kamoa and Lulua Rivers and associated streams. The area is vegetated predominantly by Central Zambezian Miombo woodland with riparian forest, grassland and wetlands. No endangered migratory animal species or plant species threatened with extinction were identified within the Project area during surveys. The Project lies just north of the Zambezi–Congo drainage divide.

1.5 History

The Kamoa-Kakula Project area was first subject to regional exploration by the Tenke Fungurume Consortium (comprising Amoco, Charter, Mitsui, BRGM and L. Tempelsman, operating as SIMZ) during 1971–1975, including a helicopter-supported stream-sediment sampling programme completed in 1971. No sample location data from this programme is available for the current Project area.

Ivanhoe Mines acquired its significant ground position, including the current permit areas, in 2003. Subsequent work programs have included data compilation, satellite imagery acquisition, geological mapping, geochemical sampling (stream sediment and soil), airborne geophysical surveys, and aircore, reverse circulation and diamond core drilling.

A maiden Mineral Resource estimate for the Kamoa deposit was prepared by AMEC (now Wood plc) in 2009, with updates in 2010, 2011, 2012, 2013, 2016, 2017, 2018, 2019, and 2020. Preliminary Economic Assessments (PEAs) for the Kamoa deposit were completed in 2012, 2013, 2016, and 2017. The Kakula deposit was first included in a resource estimate in the Kamoa 2016 Resource Technical Report filed in November 2016, and a Kakula standalone PEA was filed in January 2017.

Key milestones in the progression of study work include the Kamoa 2016 Pre-feasibility Study (establishing initial Mineral Reserves for Kansoko at 3 Mtpa), the Kamoa 2017 PFS (increasing the mining rate to 6.0 Mtpa and updating the Kansoko Mineral Reserve), the Kamoa-Kakula 2017 Development Plan (filed January 2018, increasing combined production assumptions to 12.0 Mtpa), and the Kamoa-Kakula Integrated Development Plan 2019 (establishing initial Kakula Mineral Reserves and considering an 18 Mtpa expansion scenario). The Technical Report was the Kamoa-Kakula Integrated Development Plan 2020 (effective date October 2020) reported Mineral Reserves for both the Kamoa and Kakula deposits and included a PEA considering a 19 Mtpa plant expansion scenario. The most recent preceding Technical Report was the Kamoa-Kakula Integrated Development Plan 2023 which reported Mineral Reserves updated Mineral Reserves and a PEA describing an expansion to 23.5 Mtpa.

This current Technical Report supersedes all previously reported Mineral Reserve and Mineral Resource estimates for the KCSA deposits.

1.6 Geological setting and mineralization

The mineralization identified to date within the Project is typical of sediment hosted stratiform copper deposits. The Kamoa-Kakula mineralization, 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 mineralization 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 neighboring 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 oxidized 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 mineralized stratigraphic sequence at the base of the diamictite comprises several interbedded units that host the copper mineralization. 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) mineralization. Most of the higher-grade mineralization 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, mineralization 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 mineralization, 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 mineralization is characterized by chalcopyrite and bornite dominant zones. There is significant pyrite mineralization in the KPS above the mineralized horizon that could possibly be exploited to produce pyrite concentrates for sulphuric acid production.

At the Kakula deposit, a deeper basinal setting has resulted in significant thickening of the diamictite basal units with the development of several interbedded siltstone units.

Mineralization is concentrated within a basal siltstone layer occurring just above the Roan (R4.2) contact. From the base of mineralization upward, the hypogene copper sulphides in the mineralized sequence are zoned with chalcocite (Cu2S), bornite (Cu5FeS4), and chalcopyrite (CuFeS2), with chalcocite being the dominant mineral. At Kakula, mineralization thicknesses at a 1.0% Cu cut-off grade range from 2.9–42.5 m.

Copper mineralization 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 sulphide enrichment down dip in the supergene zones. Although high-grade, these supergene zones are relatively narrow and localized. Hypogene mineralization forms the dominant mineralization style. Hypogene mineralization occurs at depths as shallow as 30 m. All three styles of mineralization occur at Kamoa; at Kakula all the mineralization occurs well below the surface and is hypogene.

1.7 Exploration

Exploration was undertaken in the current Project area by the Tenke Fungurume Consortium between 1971–1975. Although a localized 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 programs. 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 mineralization.

Integration of the geophysical programme results with the geological models supported detailed exploration targeting within the highly prospective Kamoa-Kakula exploitation license 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 mineralization to assist in locating additional targets.

In the opinion of the Qualified Person, the exploration programs 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 mineralization around known dome complexes.

Anomalies generated by geochemical, geophysical, and drill programs to date support that additional work is warranted in the Project area.

1.8 Drilling

Since commencement of drilling in May 2006, a total of 2,898 core holes (760,614.8 m) had been completed at the Kamoa-Kakula Project as of 2 December 2025. Aircore and RC drilling were utilized during early-stage exploration for anomaly follow-up; data from these hole types are not used for resource estimation. The Kamoa Mineral Resource estimate (January 2020) was based on 998 drillhole intercepts totaling 288,140.7 m, while the December 2022 Kakula Mineral Resource estimate incorporated 645 drillhole intercepts totaling 246,799.5 m.

Core holes were drilled predominantly at PQ size (85 mm), reducing to HQ (63.5 mm) and NQ (47.6 mm) where required. Geological logging was conducted electronically using acQuire software since 2012, with all core photographed wet and dry prior to sampling. Average core recovery was 95% at Kamoa and 94% at Kakula within the mineralized zones. Collar surveys were performed using differential GPS accurate to within 20 mm, and downhole surveys were conducted at maximum 50 m intervals using Single Shot and Reflex Multi Shot instruments at Kamoa, and at 3–6 m intervals using Reflex Multi Shot or Gyro instruments at Kakula.

As of 2 December 2025, 164 post-estimation drillholes (48,579 m) had been completed at Kamoa and 107 at Kakula (59,887 m) for infill and resource expansion purposes. The QP considers that the post-estimation drilling has no material effect on the overall tonnages or average grades of the current Mineral Resource estimates. The quantity and quality of drilling data are considered sufficient to support Mineral Resource estimation.

1.9 Sample preparation, analysis, and security

Whole core is logged by the geologist on major lithological intervals, until mineralized 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 mineralized or material expected to be mineralized 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 mineralization.

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 accreditations.

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 pulverized to >90% - 75 µm, using the LM2 puck and bowl pulverizers. 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 analyzed 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 mineralization at Kakula being considered to be hypogene.

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

1.10 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 QP and has found no material issues with the data as a result of these audits.

In the opinion of the QP, the data verification programs undertaken on the data collected from the Project support the geological interpretations. The analytical and database quality and the data collected can support Mineral Resource estimation.

1.11 Mineral processing and metallurgical testing

The Kamoa (Kamoa / Kansoko) resource has a long history of metallurgical testwork undertaken by various parties, focusing on the metallurgical characterization and flowsheet development for the processing of hypogene and supergene copper ores. These investigations culminated in the development of the IFS4a flowsheet, which supported the Kamoa Pre-Feasibility Study (PFS) completed in March 2016.

In 2016, Kamoa Copper SA discovered the Kakula deposit, which is characterized by significantly higher copper head grades compared to the Kamoa deposit. Consequently, the Kakula project was fast-tracked. Metallurgical testwork on the Kakula deposit commenced in 2016, with subsequent programs supporting flowsheet development and optimization.

The Kakula Phase 1 and Phase 2 concentrators were successfully commissioned in 2021 and 2022, respectively, and have since ramped up to design throughput. Both Kakula Phase 1 and Phase 2 circuits are currently undergoing modifications aimed at improving metallurgical recovery, with commissioning of the upgraded circuits scheduled for completion by the end of Q2 2026. Testwork for the recovery optimization programme was initiated at the site metallurgical laboratory and subsequently advanced by Xstrata Process Support (XPS) and Zijin Laboratories.

The Kamoa complex has also undergone expansion through its phased development plan. The Phase 3 concentrator, with a nameplate capacity of 5 Mtpa, was commissioned in June 2024 and has since ramped up, currently processing in excess of 40% of its design capacity. This development was supported by additional metallurgical testwork at XPS, including flowsheet optimization studies and a flotation variability testwork programme for material sourced from the Kansoko and Kamoa mining areas.

To support the design of the smelter, pilot scale test work was carried out by Metso–Outotec in 2002 to confirm the suitability and operating window of Kamoa and Kakula concentrates for the direct to blister flash smelting technology.

1.11.1 Kakula

Initial mineralogical and flotation testwork on the Kakula resource was conducted during 2016 and 2017 at Zijin Laboratories in China and Xstrata Process Support (XPS) in Canada. The testwork programme included two drill core samples and three composite samples, with copper head grades ranging from 3.96% to 8.19%.

Following the successful preliminary testwork results, additional drill core material was evaluated during 2017 and 2018 as part of a pre-feasibility study testwork campaign. This phase of work focused on flowsheet optimization and included a comprehensive suite of metallurgical investigations. These comprised mineralogical studies, comminution parameter determination, flotation flowsheet optimization, high-pressure grinding rolls (HPGR) testwork, concentrate and tailings thickening, filtration testing, bulk material flow characterization, comminution variability testing, and flotation variability testwork.

In March 2019, Kamoa Copper SA commissioned XPS to conduct a mini pilot plant (MPP) campaign using metallurgical drill core samples. The objective of the MPP campaign was to generate representative

material for downstream testwork programs. During the campaign, the process flowsheet was adjusted as required to prioritize the production of specific samples for various testwork initiatives.

Subsequent optimization testwork was initiated at the Kamoa site metallurgical laboratory. The results indicated that additional copper recovery could be achieved through further grinding of the coarse tailings fraction, thereby improving mineral liberation. Based on these findings, Kamoa Copper SA commissioned further testwork at both Zijin Laboratories and XPS using Kakula material to evaluate recovery improvement opportunities.

The outcomes of these testwork programs resulted in modifications to the existing Kakula processing circuits. These included the incorporation of high-intensity grinding (HIG) mills and the reconfiguration of the cleaner circuit to include three stages of cleaning, aimed at enhancing both recovery and concentrate grade.

The Kamoa-Kakula MRMR Update Technical Report recovery estimate for Kakula material is based on a combination of historical flowsheet development testwork, flotation variability testwork campaigns, and benchmarked operational data from the existing concentrators, additional recovery optimization testwork program supporting (P95) project. The recovery model incorporates results from both XPS and Zijin Laboratories.

The recovery models assume a final concentrate grade of 47.0% Cu, consistent with current plant performance and supported by testwork outcomes. Based on these assumptions, the average life-of-mine (LOM) copper recovery is estimated at 90.6% for a feed head grade of 3.34%.

1.11.2 Kamoa

Between 2010 and 2015, a series of metallurgical testwork programs (Phases 1 to 5) were conducted on Kamoa drill core samples. These programs focused on metallurgical characterization and flowsheet development for the processing of hypogene and supergene copper ores. During this period, continued resource expansion resulted in significant changes to mine schedules and associated processing strategies. As updated mine plans indicated that supergene mineralization accounted for less than 10% of the orebody, the focus of metallurgical testwork shifted primarily to hypogene ores.

The outcomes of these early-phase testwork campaigns provided the basis for the development of a Modified Flotation (MF2)-type flowsheet and established the metallurgical understanding required to support the 2012 PEA and subsequent technical reports, culminating in the Kamoa 2017 PFS.

In preparation for both the Kamoa 2016 PFS and the increased capacity scenario evaluated in the 2017 PFS, Phase 6 samples were selected to represent the early years of the Kamoa mine schedule. Metallurgical evaluation of these samples was conducted during 2014 and 2015 at XPS. It is noted that several Phase 2 and Phase 3 samples remain relevant to the current Kansoko mine schedule.

The Phase 6 testwork programme resulted in the development of the IFS4a flowsheet, which was subsequently confirmed as the preferred processing route for Kansoko material. This flowsheet was specifically tailored to the fine-grained nature of the Kamoa ore.

In 2018, XPS evaluated the performance of the Kamoa Phase 6 signature plot composite sample using the Kakula PFS flowsheet, with the objective of comparing metallurgical performance against the established IFS4a flowsheet. The composite sample achieved a final copper recovery of 86.6% at a concentrate grade

of 36.2% Cu and 13.0% SiO_2 on the Kakula flowsheet. This performance was inferior to that achieved using the IFS4a flowsheet, which delivered a copper recovery of 89.3% at a concentrate grade of 36.7% Cu and 9.1% SiO_2.

The observed reduction in recovery was attributed primarily to sample ageing effects, which were further confirmed through subsequent testwork evaluating the impact of mine water on flotation response.

Differences in metallurgical performance between the Kamoa and Kakula flowsheets were attributed to key flowsheet design variations, including:

- Improved performance in the Kakula rougher–scavenger and high-grade cleaning circuits, driven by enhanced aeration strategies and the use of additional collectors, resulting in a reduction in copper losses to rougher tailings (from 5.6% to 4.8%).
- Reduced performance in the Kakula scavenger circuit due to repositioning of the regrind stage, which increased copper losses in the scavenger cleaner and scavenger recleaner tailings (from 5.0% to 8.6%).

Despite these differences, the testwork demonstrated that Kamoa and Kakula ores exhibit similar metallurgical behavior, supporting the application of broadly comparable flotation flowsheet configurations across both ore bodies.

Further metallurgical testwork programs were subsequently conducted by Kamoa Copper SA in collaboration with XPS. These included flotation variability testwork campaigns aimed at further characterizing material from key mining areas, including Kansoko, Kamoa 1, and Kamoa 2, as feed sources for the Kamoa Phase 3 concentrator.

The Kamoa-Kakula MRMR Update Technical Report incorporates the treatment of Kamoa material using the IFS4a flowsheet. Recovery estimates for Kamoa material are based on historical flowsheet development testwork, Phase 3 flotation variability testwork, and operational data from the Kamoa Phase 3 concentrator for both supergene and hypogene ores.

The recovery model assumes final concentrate grades of 45.0% Cu for supergene ore and 33.0% Cu for hypogene ore. Based on these assumptions, the average LOM copper recovery is estimated at 83.1%, producing a concentrate grading approximately 35% Cu from a feed grade of 2.46% Cu.

1.12 Mineral Resource estimate

The Kamoa and Kakula resource models are based on the same 3D estimation methodology, and are both controlled utilizing a combination of stratigraphy and mineralized envelopes. For both Kamoa and Kakula, the mineralized envelope was defined using an approximate 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 mineralization. This method aligns the top, middle and bottom of the mineralized intervals horizontally for variography and grade estimation using ordinary kriging (OK). This preserves the important vertical grade profile and mineralogical zonation to allow vertical optimization during mine design. To adjust for local changes in the trend of the mineralization laterally, geological controls were used to locally adjust the search orientations during estimation using dynamic anisotropy.

The deposit remains open laterally, and the southern parts of the Kamoa-Kakula exploitation license area are virtually untested.

1.13 Kamoa-Kakula Combined Mineral Resource statement (on 100% Project Basis)

Table 1.1 Kamoa & combined Kakula: Indicated & Inferred Mineral Resource (on 100% Project Basis)

Deposit	Category	Tonnage (Mt)	Copper grade (%)	Contained copper (Mt Cu)
Kamoa	Indicated	750	2.73	21
	Inferred	235	1.7	4.0
Kakula	Indicated	523	2.53	13
	Inferred	75	2.1	1.2
	Inferred Pillars	26	3.5	0.9
Total Kamoa & Kakula	**Indicated**	**1,272**	**2.65**	**35**
	Inferred	**336**	**1.8**	**6.1**

Notes:

- Mineral Reserves (“reserves”) and Mineral Resources (“resources”) have been estimated as of 31 December 2025 in accordance with National Instrument 43-101 - Standards of Disclosure for Mineral Projects (NI 43-101) as required by Canadian securities regulatory authorities.
- For 31 December 2025 the long-term copper price used for estimating Mineral Resources is $6/lb.
- 1% total copper (TCu) cut-off grade has been used to report the Mineral Resource.
- Reported Mineral Resources contain no allowances for hanging wall or footwall contact boundary loss and dilution. No mining recovery has been applied.
- The Mineral Resource for Kakula was depleted to account for annual production and losses due to unextractable pillars and inaccessible areas
- Mineral Resources are reported inclusive of Mineral Reserves.
- Measured and indicated Mineral Resource estimates of grade and proven and probable Mineral Reserve estimates of grade for Cu % are reported to two decimal places.
- All inferred Mineral Resource estimates of grade for Cu % are reported to one decimal place.
- All Mineral Resource estimates of ore tonnes, copper grade and copper tonnes have been rounded to reflect the imprecise nature of the estimates for each classification category, therefore totals may not appear to sum correctly due to rounding.
- Jeremy Witley, Pr.Sci.Nat SACNASP, FGSSA of The MSA Group (Pty) Ltd estimated the Mineral Resources. The 2025 Mineral Resource was estimated from the non-depleted 2023 Mineral Resource estimate, with an effective date of 31 December 2022, and depleted to account for annual production up until 31 December 2025, as well as geotechnical losses incurred during 2025. The 2025 Mineral Resource has an effective date of 31 December 2025.
- The non-depleted 2023 Mineral Resource estimate has an effective date of 31 December 2022 and is documented in the Kamoa-Kakula Technical Report dated 16 March 2023. The cut-off date for drill data at Kamoa is 20 January 2020. The cut-off date for the drill data at Kakula is 20 July 2022, with the assay table updated as of 13 December 2022.
- Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.
- Mineral Resources are reported on a 100% basis. Ivanhoe Mines attributable ownership is 39.6% of Kamoa-Kakula.

1.14 Factors which may affect the Mineral Resource estimates

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

- Drill spacing:
  - The drill spacing at the Kamoa and Kakula deposits is insufficient to determine the effects of local faulting on lithology and grade continuity assumptions. Local faulting and steep dips around growth faults can disrupt productivity.
  - Delineation drill programs at the Kamoa deposit will have to use a tight (approximately 50 m) spacing to define the boundaries of mosaic pieces (areas of similar stratigraphic position of SMZs) in order that mine planning can identify and deal with these discontinuities. Mineralization at Kakula appears to be more continuous compared to Kamoa.

- Assumptions used to generate the data for consideration of RPEEE 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.
- Metallurgical recovery assumptions at Kamoa:
  - Variability test work 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 mineralization 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 mineralization currently identified at Kakula that requires a dedicated recovery model separate from the hypogene recovery prediction method.
- Commodity prices and exchange rates.
- These will fluctuate over the life of the project, which may impact future assessments of cut-off grade and RPEEE.
- Cut-off grades:
  - Cut-off grades are impacted by production and sales costs, metallurgical recoveries, commodity prices, and exchange rates, all of which are assumed for the purposes of Mineral Resource declaration.
- Mineral Resources in and around mature extraction area of Kakula Mine:
  - Portions of the area outside of the mature extraction zone and estimated barrier pillar that have not reached approximately 70% extraction remain in the Mineral Resource statement. While it is assumed that some of the area will be extractable, and therefore Mineral Resources remain in the area for possible future mine planning, it remains uncertain what proportion can be extracted.

1.15 Mineral Reserve estimate

The Mineral Reserve estimate for the Kamoa-Kakula mining complex was reviewed and approved by Karl van Olden, Global Lead – Underground Mining, AMC Consultants, a Qualified Person as defined under National Instrument 43-101. The effective date of the Mineral Reserve estimate is 31 December 2025.

The Mineral Reserve estimate is shown in Table 1.3. The Kamoa-Kakula Mineral Reserve is reported on a 100% basis.

Mineral Reserves are reported in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019).

The estimate uses economic and physical modifying factors, the latest Mineral Resource and geological models (as described in Section 14), geotechnical and hydro-geological inputs, and metallurgical processing recovery relationships. The Mineral Reserve is supported by work completed to a PFS level of confidence, consistent with NI 43-101 requirements for Mineral Reserve reporting. All deposits are supported by mine plans at a minimum PFS level of confidence, as described in Section 16.

Mine designs incorporate the geotechnical guidelines developed following the review of the May 2025 geotechnical event at the Kakula mine, including revised block span dimensions, inter-block pillar requirements, backfill sequencing controls and extraction ratio limitations, as described in Section 16.

Mineral Reserves have been estimated for the portions of the Kakula deposit that are unaffected by the geotechnical event in May 2025 only. Areas affected by the May 2025 geotechnical event have been excluded from the Mineral Reserve estimate pending completion of detailed underground rehabilitation and mine design studies sufficient to demonstrate safe and economically viable extraction.

Mining cut-off grade scenarios were evaluated between 1.5%TCu and 2.0%TCu. The cut-off grade applied to each individual mining area has been based on economic cut-off grade, practical mining parameters, Mineral Resource and geology characteristics and spatial considerations. Mineral Reserves have been evaluated using a long-term copper price of US$4.50/lb which the Qualified Person considers reasonable for the purposes of demonstrating economic viability. As described in Section 21, all costs used to assess economic viability are expressed in Q1 2026 US dollars. No escalation has been applied.

Mineral Reserves incorporate mining dilution and mining recovery factors applied during mine design. Dilution factors range from 0% to 20% and mining recovery factors range from 85% to 95%, varying by mining method and deposit, as described in Section 16. Pillar recovery at the end of mine life has been assumed at 30% of total pillar inventory, subject to prevailing geotechnical conditions at the time of extraction. All remnant pillars from the Kakula mature extraction zone have been excluded from any inventory at end of mine life.

Metallurgical recovery varies by deposit and by ore type. Recoveries applied are based on test work results as described in Section 13.

Mineral Reserves are reported on a dry tonne basis, and grades are reported as total copper (%TCu) as delivered to the primary, run-of-mine crushing facilities.

Table 1.2 Kamoa-Kakula Complex - Mineral Reserves Summary, 31 December 2025 (100% Basis)

Deposit	Proven			Probable			Proven and Probable		
	Ore (Mt)	Copper (%)	Copper (Contained Mt)	Ore (Mt)	Copper (%)	Copper (Contained Mt)	Ore (Mt)	Copper (%)	Copper (Contained Mt)
Kakula	-	-	-	51.3	3.94	2.02	51.3	3.94	2.02
Kakula West	-	-	-	84.4	2.98	2.52	84.4	2.98	2.52
Konsoko Sud	-	-	-	32.6	2.71	0.88	32.6	2.71	0.88
Kamoa 1	-	-	-	103.9	2.71	2.82	103.9	2.71	2.82
Kamoa2	-	-	-	78.0	2.59	2.02	78.0	2.59	2.02
Kamoa 3	-	-	-	57.6	2.41	1.39	57.6	2.41	1.39
Kamoa 4	-	-	-	42.7	2.46	1.05	42.7	2.46	1.05
Kamoa 5	-	-	-	8.6	2.66	0.23	8.6	2.66	0.23
Kamoa 6	-	-	-	7.2	2.74	0.20	7.2	2.74	0.20
Total	**-**	**-**	**-**	**466**	**2.82**	**13.13**	**466**	**2.82**	**13.13**

Notes:

- Mineral Reserves (“reserves”) and Mineral Resources (“resources”) have been estimated as of 31 December 2025, in accordance with National Instrument 43-101 - Standards of Disclosure for Mineral Projects (“NI 43-101”) as required by Canadian securities regulatory authorities.
- For 2025 the long-term copper price used for calculating Mineral Reserves and economic mine plan analysis is $4.50/lb. The long-term copper price used for calculating Mineral Resources is $6.00/lb.
- Realization costs include refining and treatment charges, deductions and payment terms, blister and concentrate transport, metallurgical recoveries, and royalties.
- Cut-off grades applied to the Mineral Reserve are between 2.0% TCu and 1.5% TCu. The varying characteristics of each deposit, and the intention of maintaining reliable mining parameters and geotechnical controls has resulted in each scenario applying both a minimum economic cut-off, practical mining parameters and spatial considerations to differentiate between mined material considered to be ore or waste.
- In confirming the Mineral Reserves for Kamoa & Kakula, a reserve test has been undertaken, to verify that the future undiscounted cash flow from reserves is positive. The cash flow ignores all sunk costs and only considers future operating and closure expenses as well as any future capital costs.
- Metallurgical recovery for each Concentrator is defined by the application of a recovery algorithm. The metallurgical recovery is 87.98% for the Kakula and Kamoa concentrators (Mineral Reserve life-of-mine plan average).
- Smelter recovery is 98.5%.
- Mineral Reserve tonnage and grade estimates include apportionment for dilution and recovery.
- Mineral Reserves reported above are inclusive of Mineral Resources and are not additive.
- Totals may not appear to sum correctly due to rounding.
- All Mineral Resource and Mineral Reserve estimates of tonnes, Cu tonnes and pounds are reported to the second significant digit.
- Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.
- Measured and indicated Mineral Resource estimates of grade and proven and probable Mineral Reserve estimates of grade for Cu % are reported to two decimal places.
- All inferred Mineral Resource estimates of grade for Cu % are reported to one decimal place.

1.15.1 Mineral Reserve risk factors

In the opinion of the QP, the Mineral Reserves are subject to the type of risks that are common to underground mining operations and may be materially affected by the following risk factors:

- Changes in realized metal prices from what was assumed.
- Changes to the mining costs, processing and G&A costs used to calculate the cut-off grade.
- Changes in local interpretation of mineralization geometry or modelled continuity of mineralized zones.
- Changes to geotechnical or hydrogeological design assumptions resulting in schedule delays, increased dilution, or reduced recoveries.
- Changes to mining and metallurgical recoveries.
- Changes in the long-term assumptions relating to product payability, marketability, and penalty terms.
- Assumptions as to the continued ability to access the site, retain mineral tenure, obtain required environmental, mining, and other regulatory permits, and maintain a social license to operate.

1.16 Mining methods

1.16.1 Mining complex overview

The Kamoa-Kakula mining complex (KCSA) comprises nine underground deposits exploiting stratiform copper orebodies. The Kamoa 1 to 6 deposits are located in the central to northern extents of the property, Kansoko Sud is centrally located, and the Kakula and Kakula West deposits are in the south. The property extends approximately 30 km north–south and 20–30 km east–west. In 2026, active mining operations are underway at Kakula, Kamoa 1, Kamoa 2 and Kansoko Sud. The remaining deposits — Kamoa 3, Kamoa 4, Kamoa 5, Kamoa 6, Kakula West-East and Kakula West-West — are planned future operations supported by prefeasibility-level studies.

The life-of-mine strategy targets a sustained processing rate of 17 Mtpa across the combined concentrator and smelter infrastructure. A cut-off grade of between 1.5% and 2.0% TCu has been applied, varied by deposit based on economic and physical considerations.

1.16.2 Geotechnical setting

The orebodies are hosted within layered sedimentary sequences comprising breccias, diamictite (SDT), siltstone (SSL) and sandstone (SST), dipping between approximately 9° and 35° (average 18°) and ranging in thickness from 2.5 m to 20 m (typically 6 m). Mining depths extend from 70 m to 1,400 m below ground level, with current operations at approximately 300 m depth.

Rock mass conditions range from very poor to good across the complex, influenced by major structures including the West Skarp Fault, the Bonanza Fault, and primary depositional faults. Ground conditions are generally classified as fair to good based on Q-system ratings; however, localized zones of very poor ground are present, particularly associated with fault zones and orebody breccia units. Geotechnical assessments have been completed by AMC Consultants, Beck Engineering, OHMS and others, incorporating rock mass classification (RMR, Q, GSI), laboratory testing, structural logging and numerical modelling.

In May 2025, a significant geotechnical event occurred at the Kakula mine, resulting in substantial damage to underground excavations and flooding. Detailed investigation and numerical modelling identified cascading failure through mining precincts as the cause, exacerbated by high extraction ratios, wide block

spans and limited stabilization pillars. The findings of this investigation have been incorporated into revised mine design parameters applied across all deposits. Key design changes include reduced block spans (maximum 300 m × 300 m), increased inter-block pillar dimensions, mandatory backfilling controls, staged extraction sequences and enhanced geotechnical monitoring requirements.

Ground support is designed based on Q-system classifications and the All Kamoa Copper Support Standards (2025), with primary support comprising split sets, mesh, cable bolts and fibrecrete applied according to ground conditions. All mine designs incorporate systematic cablebolting at intersections and in wide spans.

1.16.3 Mining methods

Four underground mining methods are deployed across the complex, selected based on orebody dip, thickness, depth and rock mass conditions:

- Drift and Fill (D&F): Applied in flat to shallow dipping areas (0°–20°) with orebody thickness 2.5–20 m. Primary ore drives are developed at 6 m × 6 m, retreated using longhole stripping to a maximum stoping width of 7.5 m, and backfilled with cemented paste fill. Primary and secondary extraction sequences are employed with backfill curing of 30 days prior to secondary extraction. This is the predominant method at Kakula, Kamoa 1, Kamoa 2, Kansoko Sud and Kakula West-East.
- Cut and Fill (C&F): Applied in moderately dipping areas (30°–40°) with mining thickness 5–15 m. Ore is extracted in bottom-up lifts of 5 m, with paste fill placed after each lift. Applied at Kakula West-West and parts of Kamoa 3.
- Room and Pillar (R&P): Applied in flat dipping areas (0°–5°) with mining heights below 6 m. Production panels of approximately 150 m × 150 m are separated by chain pillars. Panel-scale extraction ratios of approximately 75% are achieved, with an overall mine recovery assumption of approximately 70% including regional pillars. Applied at Kamoa 2 (flat areas), Kamoa 4, Kamoa 5 and portions of Kamoa 6.
- Longhole Stoping (LHS): Applied where orebody dip exceeds approximately 55°, with stope heights of 15 m and lengths limited to 15 m in fair to poor ground. Cemented paste backfill is required. Applied at the Bonanza zone within Kamoa 4 and at Kakula West-West.

1.16.4 Mine ventilation and cooling

Ventilation systems have been designed for each deposit based on diesel exhaust dilution as the primary design criterion. Design criteria include a maximum workplace wet-bulb temperature of 27.5°C and a minimum diesel dilution rate of 0.063 m^3/s per rated kW. Primary ventilation is provided by surface exhaust fan stations (900 kW trifurcated axial flow, or 560 kW bifurcated for lower-flow mines), with auxiliary forcing ventilation to development faces. Refrigeration is required at Kakula (18 MWc), Kamoa 1 (17 MWc) and Kakula West-West (9 MWc), and is phased in line with the mining schedule.

1.16.5 Dewatering

Groundwater management is a significant operational challenge across the complex. The principal aquifer is the Roan Sandstone (R4.2 unit) in the footwall, recharged from surface and approximately 1,000 m thick. The West Scarp Fault and associated splay faults are major conduits for water ingress. Dewatering systems are designed around a primary system of multistage pump stations capable of pumping up to 2,000 l/s per station to surface through dewatering boreholes, supported by secondary face and transfer pump systems. Peak dewatering requirements range from 120 l/s (Kamoa 6) to 11,600 l/s (Kakula over life-of-mine). Kakula

West-East incorporates a dedicated drainage horizon below the orebody footprint to manage anticipated peak inflows of approximately 4,500 l/s.

1.16.6 Production profile

The combined Mineral Reserve production profile for the nine KCSA deposits is summarized below. Total production of approximately 467 Mt at 2.8% Cu (13.1 Mt of contained copper) is planned, with the operation targeting a sustained throughput of 17 Mtpa. The Mineral Reserve plan achieves steady production of 17Mtpa between 2028 and 2050, with a tapering tail of reducing production as deposits are depleted.

Table 1.3 Mineral Reserve production profile for KCSA deposits

Deposit	Ore (Mt)	Grade (%Cu)	Cu (Mt)	Peak (Mtpa)	Life (yrs)
Kamoa 1	103.9	2.7	2.8	5.5	26
Kamoa 2 Central & East	57.0	2.6	1.5	3.3	27
Kamoa 2 West	21.0	2.5	0.5	2.8	12
Kansoko Sud	32.5	2.8	0.9	2.6	26
Kakula	51.3	3.9	2.0	8.0	8
Kakula West-East	52.5	2.5	1.3	3.0	37
Kakula West-West	31.9	3.8	1.2	2.2	30
Kamoa 3	57.6	2.4	1.4	3.5	21
Kamoa 4	43.0	2.5	1.1	3.9	17
Kamoa 5	8.6	2.7	0.2	1.5	12
Kamoa 6	7.2	2.7	0.1	1.3	9
Total	**466**	**2.8**	**13.1**		

1.17 Recovery methods

1.17.1 Overview

The Kamoa-Kakula 2026 Mineral Resource and Mineral Reserve Update considers a total concentrator production capacity of approximately 17 Mtpa. This is based on the modified capacity of the Kakula Phase 1 and Phase 2 concentrators of 10.5 Mtpa, together with the achieved operating capacity of approximately 6.5 Mtpa from the Kamoa Phase 3 concentrator. Concentrate produced at both plants is processed at the on-site Kakula copper smelter to produce copper anode. Process plant design was undertaken by DRA Projects (Pty) Ltd for the concentrators and China Nerin Engineering Co. Ltd for the smelter.

1.17.2 Kakula concentrator — Project 95

The Kakula concentrator complex comprises two identical processing modules with a combined design throughput of 10.5 Mtpa (5.25 Mtpa per module). The complex is currently undergoing a series of modifications under the "Kakula optimization Project (P95)" programme, aimed at increasing copper recovery from its current level to 95% while maintaining the design processing rate.

The Kakula concentrator flowsheet incorporates a conventional comminution and flotation circuit, designed to maximize copper recovery while producing a high-grade concentrate.

Run-of-mine (ROM) ore is first stockpiled and then processed through cone crushers operating in closed circuit with vibrating screens to produce a crushed product with 100% passing 50 mm. The crushed material is stockpiled prior to being fed to a High-Pressure Grinding Rolls (HPGR) circuit operating in closed

circuit with wet screening, producing a product with a P80 of 4.5 mm. The HPGR product is gravity-fed to the milling circuit.

The milling circuit comprises two stages of ball milling in series, operating in closed circuit with cyclone clusters to achieve a target grind size of 80% passing 53 µm. A flash flotation circuit is incorporated within the milling circuit to recover liberated chalcocite and minimize overgrinding.

The milled slurry is classified, with the coarse and fine fractions directed to separate rougher flotation circuits. These circuits produce a high-grade (fast-floating) rougher concentrate and a medium-grade (slow-floating) scavenger concentrate. The rougher concentrate is upgraded in a low-entrainment, high-grade cleaner circuit to produce a final high-grade concentrate.

The scavenger concentrate, together with tailings from the high-grade cleaner stage and coarse rougher tailings, is directed to a regrind circuit, where the material is ground to a P80 of approximately 10 µm. The regrind product is further upgraded through a reconfigured three-stage scavenger cleaning circuit. The scavenger recleaner concentrate is combined with the high-grade cleaner concentrate to form the final concentrate product.

The final concentrate is thickened and filtered prior to either bagging for shipment or transportation to the Kakula smelter complex. Tailings from the scavenger and cleaning circuits are thickened and directed either to the backfill plant or the tailings storage facility.

1.17.3 Kamoa concentrator

The Kamoa concentrator flowsheet is broadly similar to that of the Kakula concentrator, with the primary distinction being the positioning of the regrind circuit within the flotation flowsheet.

Run-of-mine (ROM) ore is stockpiled and processed through primary cone crushers operating in closed circuit with vibrating screens to produce a crushed product with 100% passing 50 mm. The crushed material is stockpiled prior to being fed to the High-Pressure Grinding Rolls (HPGR) circuit, which operates in closed circuit with wet screening to produce a product with a P80 of approximately 4.5 mm. The HPGR product is then gravity-fed to the milling circuit.

The milling circuit consists of two stages of ball milling in series, operating in closed circuit with cyclone clusters to achieve a target grind size of 80% passing 53 µm. The milled slurry is then pumped to the rougher and scavenger flotation circuits, where separation of high-grade (fast-floating) rougher concentrate and medium-grade (slow-floating) scavenger concentrate is achieved.

The rougher concentrate is upgraded in a low-entrainment, high-grade cleaner circuit to produce a high-grade concentrate. The scavenger concentrate, together with the tailings from the high-grade cleaner stage, is directed to the regrind circuit, where it is ground to a P80 of approximately 10 µm prior to further upgrading.

The regrind product is combined with scavenger recleaner tailings and processed through the scavenger cleaner circuit. This circuit is designed with flexibility to produce either a medium-grade concentrate or a low-grade concentrate, depending on feed conditions. The medium-grade concentrate reports directly to the final concentrate stream, while the low-grade concentrate is further upgraded in a low-entrainment scavenger recleaner stage.

The scavenger recleaner concentrate is combined with both the high-grade cleaner concentrate and the medium-grade scavenger cleaner concentrate to form the final concentrate product. The final concentrate is thickened and filtered prior to transport to the Kakula smelter complex.

Tailings from the scavenger and cleaning circuits are combined and thickened before being directed to either the backfill plant or the tailings storage facility.

1.17.4 Kakula copper smelter

The Kamoa-Kakula 500 ktpa copper smelter employs Direct-to-Blister Flash (DBF) smelting technology, licensed from Metso, to process blended concentrates from the Kakula and Kamoa concentrators. The DBF concept was first identified as suitable for the project in 2012 and has remained central to all subsequent technical studies, reflecting the consistent and predictable mineralogy of the ore body. Basic engineering was awarded to China Nerin Engineering in late 2021, with the final EPCM contract following in 2022. Construction and cold commissioning were completed in May 2025, with hot commissioning commencing in December 2025 following the resolution of power stability issues through the installation of a 60 MW uninterruptible power supply. First anode copper was produced in December 2025, and the smelter is currently ramping up towards its nameplate capacity.

The smelter is designed to treat approximately 1.2 million tonnes per annum (Mtpa) of blended concentrate to produce 500 ktpa of copper anode grading greater than 99.66% Cu, together with approximately 783 ktpa of sulphuric acid as a co-product. The process flowsheet comprises concentrate blending and steam drying, the DBF furnace, a 30 MVA six-electrode rectangular electric Slag Cleaning Furnace (SCF), three 660-tonne anode refining furnaces with twin anode casting machines, a modified double-contact double-absorption (DCDA) sulphuric acid plant, and a conventional comminution and flotation slag treatment plant. Oxygen is supplied by a vacuum pressure swing adsorption (VPSA) plant, and waste heat from the DBF off-gas and the Acid Plant is recovered via a waste heat boiler to generate steam for concentrate drying and approximately 8 MW of electrical power. The smelter is designed to operate 7,400 hours per annum (84.5% availability).

A critical design constraint is the Kamoa : Kakula concentrate blend ratio. The high-energy chalcopyritic nature of Kamoa concentrate limits its proportion in the feed blend to a maximum of approximately 70%, beyond which the exothermic heat balance of the DBF becomes unmanageable, acid plant capacity is exceeded, and the slag treatment plant design margin is insufficient to handle the increased slag volumes. Conversely, the DBF is well suited to the low-energy, high-bornite and chalcocite Kakula concentrate.

As of the effective date of this report, several operational issues have been identified during initial commissioning and ramp-up. The VPSA oxygen plant has not achieved design production rates, necessitating the installation of additional modules. A launder blockage issue at the entry point to Anode Furnace No. 3, attributed to its side-entry configuration, requires the installation of additional burners. Furthermore, the standby ladle arrangement for transferring blister copper from the SCF to Anode Furnaces No. 1 or No. 2 has proven inadequate due to excessive heat loss and skull formation in the cast iron ladles; remediation options under consideration include installation of a fourth smaller anode furnace or the procurement of insulated transfer ladles with reheat burners. Notwithstanding these early-operation challenges, smelter ramp-up has progressed smoothly, with current production at approximately 70% of nameplate capacity.

1.18 Project Infrastructure

The infrastructure required to support LOM operations comprises a combination of on-site and off-site facilities, strategically developed to ensure safe, efficient, and sustainable mining activities. On-site infrastructure is established at each mine's boxcut during the mine access development phase and includes essential buildings, workshops, and service infrastructure required for day-to-day operations.

Off-site infrastructure is developed across the orebody footprint over the LOM and supports broader mining functions. This includes ventilation systems and associated auxiliary services necessary for underground production, ensuring adequate airflow, environmental control, and operational continuity.

Collectively, the infrastructure scope encompasses critical systems such as access and haul roads, power supply and reticulation networks, water supply and sanitation systems, stormwater and mine dewatering management, and acid water management (where applicable). In addition, it includes ventilation infrastructure, surface and underground buildings, backfilling systems, and tailings delivery pipelines with associated servitudes. This integrated infrastructure framework is designed to support operational efficiency, environmental compliance, and long-term sustainability across all mining activities.

A bottom-up estimation methodology was applied to determine the electrical power requirements and Maximum Demand (MD) for both surface and underground installations. The MD is defined as the maximum electrical demand, measured in kVA, over a 30-minute period. Load estimates were developed using detailed load lists compiled per area in MS Excel, incorporating all equipment power requirements as defined in the Mechanical Equipment List (MEL). The MEL developed for the Kamoa Life-of-mine NI 43-101 MRMR Update served as the primary input for this assessment.

Mechanical loads were adjusted using appropriate de-rating, diversity, and utilization factors to reflect realistic operating conditions and to derive expected running loads. Furthermore, mining and production schedules were applied to these running loads to establish an MD load profile in kW, ensuring that the projected power demand accurately represents operational scenarios over the LOM.

Key underground infrastructure and systems required to support underground mining operations are focused on water management, material handling, utilities, and maintenance facilities to ensure safe and efficient LOM operations.

Mine water and groundwater inflows are managed through a staged dewatering system comprising sumps, transfer dams, vertical transfer dams, and multistage pumpstations. Submersible pumps transfer dirty water from working areas through a series of pumpstations to surface settling dams. Pumping systems are designed with multiple pump trains for redundancy, with high-lift multistage pumps and upstream treatment (e.g., de-gritters and separators) ensuring efficient and reliable operation.

Material handling is achieved through a combination of load-and-haul equipment and conveyor systems, enabling flexible transport of ore and waste from working faces to surface stockpiles.

Supporting utilities include compressed air supplied from surface to refuge chambers, firewater and fire protection systems for underground safety, potable water from borehole sources, and service water largely supplied through recirculated mine water. Additional systems include emulsion supply, fuel and lubricant distribution, and shotcrete delivery, all supported by appropriate environmental controls.

Maintenance is facilitated through strategically located main underground workshops equipped for major servicing and repairs, supported by satellite workshops closer to active mining areas for routine maintenance. These facilities are integrated with ventilation systems to ensure safe working conditions.

Overall, the infrastructure provides a reliable, efficient, and scalable framework to support sustained underground mining operations.

1.19 Environmental studies, permitting, and social or community impact

1.19.1 Environmental baseline studies and impact assessment

Kamoa Copper SA has undertaken a comprehensive Environmental and Social Impact Assessment (ESIA) programme spanning the initial project development phase through to subsequent staged expansions. The most recent ESIA, prepared by Knight Piésold Consulting RDC SARL, was finalized and submitted to the Directorate for the Protection of the Mining Environment (DPEM) in April 2025. The 2025 ESIA is compliant with DRC Mining Regulations (Article 463 of Decree No. 038/2003, as amended by Decree 18/024) and aligns with IFC Performance Standards. It integrates updated specialist studies including WSP (2024) tailings geochemistry and hydrogeological modelling, Airshed (2024) air emissions and dispersion modelling, SRK (2024) ambient air monitoring, WKC (2024) noise and vibration assessments, and KELKAM (2024) radiation monitoring.

The ESIA identifies a range of environmental and social impacts across construction, operation, and decommissioning phases. The overall conclusion of the ESIA does not identify any fatal flaw concerns or impacts that cannot be mitigated to acceptable levels that could materially affect Kamoa's ability to extract the Mineral Resources and reserves. Key impact areas assessed include land access and resettlement, water management and hydrogeology, acid rock drainage (ARD) and tailings management, biodiversity and sensitive ecosystems, air quality, noise and vibration, closure and financial provisioning, and social licence and community health. Multiple radiation assessments (NECSA 2013, KELKAM 2019–2024) confirm normal background radiological conditions with no enhanced risk from mining activities.

1.19.2 Waste, tailings, water, and monitoring

Tailings are managed across the Kakula TSF (Cells 1–3) and the future Mupenda TSF, both incorporating partial HDPE lining, downstream embankment construction, and engineered stormwater controls. Phase 3 tailings have been classified as non-potentially acid generating and non-radioactive. Copper and iron concentrations exceed low-risk leachability thresholds, and ongoing geochemical monitoring is maintained. Kamoa plans to co-dispose approximately 0.6 Mtpa of smelter waste (slag, gypsum, and neutralized sludge) with tailings slurry; further site-specific characterization of smelter waste is recommended to refine geochemical predictions. Hydromining of Kakula TSF Cell 1 historic tailings for reprocessing is planned.

Water management is structured around maximizing recycling, with TSFs returning an average of approximately 8,748 m^3/day (approximately 89% of water sent to the TSF) to processing plants. Acid-generating water from KPS waste and ore stockpiles is collected in lined ponds and treated by high-density sludge (HDS) processes. A comprehensive environmental monitoring programme covers surface and groundwater quality, air quality (PM_{10}, $PM_{2.5}$, SO_2, NO_2, metals), embankment stability, and seepage, with results reported to the DPEM and ACE. Post-closure monitoring is planned for a minimum of 10 years, focused on physical stability, water quality, vegetation establishment, and erosion control.

1.19.3 Permitting

The Project holds valid Exploitation Permits (PE 12873, PE 13025, PE 13026) covering all mining areas, valid through 19 August 2042. An Environmental Certificate (No. 0165/ACE/DG/LO/AIE/MK/2024) is current to July 2027, with the updated ESIA subject to DPEM approval. A Financial Guarantee for rehabilitation is maintained with annual payments, and annual surface rights and area taxes are up to date. The Cahier des Charges (social obligations agreement) has been submitted and is pending approval. Kamoa maintains a satisfactory compliance record with no significant regulatory violations or enforcement actions reported by the DPEM or ACE.

Key permitting risks include potential delays in ESIA / ESMP approval for future expansions (Mupenda TSF, Kamoa 2 Open Pit, solar farm), pending renewal of certain infrastructure and telecommunications permits, and the requirement for ongoing stakeholder consultation as expansions proceed.

1.19.4 Social and community impact

The Project's social area of influence encompasses approximately 41 communities (approximately 21,000 people) within the Mutshatsha Territory. Extensive stakeholder engagement was conducted in October 2024, with key issues raised including relocation, water access, community health, employment, and infrastructure. Resettlement Action Plans (RAPs) have been implemented in compliance with DRC Mining Code Article 281 and Mining Regulations Annex XVIII. A revised resettlement plan, informed by Q4 2024 demographic and property surveys, has been established.

Community development is supported through the Cahier des Charges, with a five-year programme covering education, health, infrastructure, and livelihoods. A Centre of Excellence provides access to tertiary education and supports local procurement. Cultural heritage assessments have identified over 70 cemeteries, 21 archaeological sites, and 25 sacred sites within the concession, with protocols in place for protection and community consultation. The ESIA's social risk assessment identifies physical and economic displacement as the highest residual risk, managed through RAPs and livelihood restoration programs. No unmitigable social risks were identified.

1.19.5 Mine closure and reclamation

A Mine Closure and Reclamation Plan (MCRP) was completed in 2023 by OMI Solutions and is framed around progressive rehabilitation during operations and a risk-informed approach to final closure. Closure objectives encompass physical and geochemical stabilization, reinstatement of land capability, support for native vegetation re-establishment, and long-term community benefit. The closure cost provision is listed in Section 21, comprising general surface rehabilitation, infrastructure decommissioning, mining aspects, surface water management, TSF rehabilitation, additional allowances, and 10 years of post-closure monitoring. The financial provision is calculated using the South African DMR rule-based methodology and is maintained through annual allocations updated to reflect changes in site conditions and project scope.

1.20 Capital and operating costs

Capital and operating cost estimates for the Kamoa-Kakula Copper Project have been prepared to at least PFS standard, drawing on contractual and quoted firm estimates as well as cost data from ongoing operations. All figures are based exclusively on the Mineral Reserve mine plan and are expressed in real US$ as at Q1 2026.

1.20.1 Capital costs

The total capital cost to implement the Mineral Reserve plan is $17.11 billion (Including Sustaining Capital), comprising mining development ($5.73B), mining infrastructure ($4.69B), mining mobile equipment fleet ($1.75B), surface infrastructure capital ($2.00B), and indirects & others ($2.92B). The dominant capital cost functions are mining development and fit-out of underground mines inclusive of portals and boxcuts, conveyors, dewatering, power, backfill, ventilation, and fixed installations. The surface infrastructure covers access roads, water management, solar farms, and tailings storage facilities. Tailings infrastructure inclusive of future lifts represents $0.73 billion of this total. No expansions of processing capacity are incorporated within the Mineral Reserve plan.

1.20.2 Operating costs

The steady-state unit cost is estimated at $145.9 per tonne milled at 17 Mtpa. The largest cost components are mining ($86.6/t), processing ($21.2/t), smelting operations ($18.0/t). Logistic costs are $12.0/t), with general and administrative, TC, RC comprising the remainder. The value of sulphuric acid production is treated as a credit. The long-term sulphuric acid price is assumed to be $350/tonne.

The QP has validated that the capital and operating cost estimates are based on actual costs and appropriate provisions for future cost trends, and considers the estimates to be at an appropriate level of detail to support the Mineral Reserve plan.

1.21 Economic analysis

This section is not required as Ivanhoe Mines is a producing issuer, as one of the principal shareholders of Kamoa Copper SA, the operator of the Kamoa-Kakula Mining Complex, the operations are currently in production, and there is no material expansion of current production planned.

An economic analysis of the Kamoa-Kakula Mining Complex has been completed using the Mineral Reserve estimates presented in this Report and the QP has verified that the outcome is a positive cash flow at a $4.50/lb assumed copper sales price which confirms the economic viability of the Mineral Reserves.

1.22 QP interpretations and conclusions

- The May 2025 Seismic event at Kakula is a watershed event for KCSA. The post event analysis has fundamentally reset mine design and operational management philosophy across all deposits. Global stability findings emphasize requirements for company governance and controls of block spans, pillar dimensions, extraction sequencing and backfill timing. The observational approach with defined decision gates before each extraction phase, is a key control. Consistent implementation and maintenance of these controls by operational management should be a focus of senior KCSA management.
- The Kamoa-Kakula mining complex is a well-established operation with integrated infrastructure comprising mines, concentrators, smelter, accommodation and infrastructure to support a world leading copper mining and beneficiation organization. The development of the operation has been accelerated and the achievements to date, to establish the business have exceeded those achieved by most peer companies. As the business transitions from a development project to a steady producing operation with a long life, the company should focus on establishing and maintaining the systems, standard practices, and management frameworks that will support reliable integrated production across the life of the operation. The QP is satisfied that the Kamoa Kalula Copper complex

is well established and is able to support the planned production levels over the Mineral Reserve mine life.

- The QP is satisfied that the mine designs and subsequent mining plans are sufficient to demonstrate physically and economically viable mining recovery.
- In the opinion of the QP, the metallurgical test work conducted for the Kakula and Kamoa deposits is sufficient for both PFS and feasibility level process design respectively. The samples tested are representative of the target mining area. The comminution characteristics are well established and have consistency across the various testing phases and across the prospective mining areas. Sufficient work has been conducted to design the regrind circuits.
- The QPs are satisfied that the existing and planned access, infrastructure, power and water supply, workforce availability and logistics are reasonably well established to support year-round mining operations.
- The QP considers that geotechnical data confidence varies across the deposits. The operating mines (Kakula, Kamoa 1, Kamoa 2, Kansoko Sud) benefit from substantial underground mapping data. Future deposits (Kamoa 3–6, Kakula West) are supported by wide-spaced drillhole data and have identified gaps in laboratory testing and in situ stress measurement. Recommendations for each deposit include infill geotechnical drilling, expanded laboratory testing programs (TCS and UTB in particular), in situ stress measurements, and ongoing underground mapping as development advances. Numerical modelling of mine designs is ongoing, and mine plans are to be updated as additional data becomes available.
- The QP considers that a perpetual challenge to mining operations will be dewatering ahead of planned underground mining activities. Substantial, proactive analysis, testing and dewatering initiatives are required in the near to medium-term to enable continued operational performance at planned rates.
- The QP is satisfied that the work undertaken to identify and quantify Environmental, Social, and Community compliance requirements and potential impacts has been appropriately addressed. Provisions to prevent, manage and mitigate these impacts have been adequately developed and funded.
- The QP has verified that the economic outcome of the Mineral Reserve mine plan is a positive cash flow at a $4.50/lb assumed copper sales price which confirms the economic viability of the Mineral Reserves.

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2 Introduction

2.1 Ivanhoe Mines Ltd.

Ivanhoe Mines Ltd. (Ivanhoe) is a mineral exploration, development and mining company, whose principal properties are located in Africa. The Ivanhoe strategy is to build a global, commodity-diversified mining and exploration company. Ivanhoe has focused on exploration within the Central African Copperbelt, and the Bushveld Complex.

Ivanhoe currently has three key assets: An interest in the Kamoa-Kakula Copper Complex; 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.

Ivanhoe owns a 39.6% interest in the Kamoa-Kakula project through a 49.5% interest in Kamoa Holding Limited (Kamoa Holding).

2.2 Terms of reference

The Kamoa-Kakula Mineral Resource and Mineral Reserve Update (MRMR Update) is an independent technical report prepared in accordance with Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for Ivanhoe Mines. The Report addresses the Kamoa-Kakula Copper Complex, operated by Kamoa Copper SA (KCSA) and located in the Democratic Republic of Congo (DRC).

KCSA is situated in the Mutshatsha territory in the Lualaba Province, DRC. The operation is located within the Central African Copperbelt, approximately 25 kilometres (km) west of the provincial capital of Kolwezi, and about 270 km west of the regional centre of Lubumbashi. The operation includes the Kamoa and Kakula stratiform copper deposits.

The following companies have undertaken work in preparation of Kamoa-Kakula MRMR Update. This work is the primary source of information and data underlying the analysis described in this Report:

- AMC Consultants: Mine design and planning, Geotechnical Engineering, Mineral Reserve estimation, cost estimation and cashflow analysis.
- The MSA Group: Geology, drillhole data validation, and Mineral Resource estimation for Kamoa and Kakula.
- DRA Global: Capital cost estimation, Process and infrastructure.
- WSP: Hydrogeology and Closure costs.
- Epoch Resources (Pty) Ltd: Tailings Storage Facility (TSF).
- PRODEO Consulting (Pty) Ltd, Smelter.
- Kamoa Copper SA: Property description and location, ownership, mineral tenure, environmental studies, permitting and social and community, operational information and marketing.
- Paterson and Cooke: Backfill

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.

Table 2.1 Qualified Person responsibilities

Qualified Person	Qualification	Sections and subsections	Site visit
Karl van Olden	FAusIMM	Portions of Section 1; Sections 2 to 6; Sections 15, 16; 18 to 24 and portions of Section 25 and Section 26	3 x visits of 5 days each - August and December 2025 and January 2026.
Jeremy Witley	Pr. Sci. Nat.	Sections 1.6 to 1.10, 1.12 to 1.14, 7 to 12 and 14, portions of Sections 25 and 26	15–22 August 2022
Tony Nyakudarika	Pr.Eng	13; 17.1 to 17.3 and parts of 21, 26.3	23 to 28 September 2025
Steve Amos	Pr.Eng	1.17.4 and 17.4	Multiple times throughout 2025, Most recently August 2025 for 5 days.
Andrew Savvas	Pr.Eng, CPEng	Section 18.10	2 to 5 March 2026

2.4 Effective dates

- The effective date of this Report: 31 March 2026.
- 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 and Kakula Mineral Resources were depleted to account for annual production and have an effective date of 31 December 2025.
- Date of the Mineral Reserve estimate for Kamoa-Kakula: 31 December 2025.
- Date of the supply of legal information supporting mineral tenure: 24 March 2026.
- All cost and cashflow estimates are based on Real, Quarter 1 2026 US Dollar money terms.

2.5 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 to QPs 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

3.1 Overview

In preparing this Technical Report, the QP has relied on information, opinions, and representations provided by the Issuer and its appointed legal and regulatory advisors with respect to certain matters that fall outside the QP's specific areas of technical expertise. This reliance is made in accordance with Section 5.2 of National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101) and the associated Companion Policy 43-101CP.

The QP has exercised professional judgment in determining which matters are appropriately delegated to other experts and has taken reasonable steps to confirm the reliability of the information provided, including review of documentation supplied by the Issuer and its advisors. However, the QP does not hold qualifications in law, regulatory compliance, or the administration of mineral rights under the laws of the DRC and accordingly has not independently verified the legal conclusions described in this section.

3.2 Legal tenure and mineral rights

The QP has relied on the Issuer to confirm the legal standing, ownership, and tenure of all mineral rights, surface rights, and associated concessions comprising the Project in the DRC. Specifically, the QP has relied upon representations and warranties made by the Issuer and, where applicable, opinions provided by the Issuer's DRC-qualified legal counsel, with respect to the following:

- The valid existence, registration, and good standing of all exploitation permits (Permis d'Exploitation, PE), research permits (Permis de Recherches, PR), and any other mineral tenure instruments applicable to the Project, as maintained under the DRC Code Minier and its implementing regulations (Règlement Minier).
- The identity of the registered holder(s) of each mineral title and the Issuer's legal right, title, and interest in and to such titles, whether held directly or through subsidiary entities incorporated under the laws of the DRC.
- That all mineral tenure instruments are free from material encumbrances, challenges, liens, disputes, or third-party claims known to the Issuer as of the effective date of this Technical Report.
- That all annual area taxes (taxe superficiaire), regulatory fees, and other charges required to maintain the mineral titles in good standing have been paid in full and on time in accordance with the requirements of the Code Minier and the Cadastre Minier (CAMI).
- That no notice of cancellation, forfeiture, suspension, or adverse action has been received from CAMI, the Ministry of Mines, or any other competent authority in respect of any mineral title comprising the Project.

The QP has reviewed copies of the relevant permit certificates, CAMI registration extracts, and title confirmation letters as provided by the Issuer. The QP has not independently conducted searches of the CAMI register or obtained independent legal opinions confirming the status of such titles and relies entirely on the Issuer's representations and the documentation provided to the QP for the purposes of the disclosures made in this Technical Report.

3.3 Currency and validity of applicable permits

In addition to mineral tenure, the QP has relied on the Issuer to confirm the currency, validity, and compliance status of all applicable operational, environmental, and regulatory permits, authorizations,

and approvals required for the conduct of exploration and / or mining activities at the Project under DRC law. This reliance encompasses, but is not limited to, the following categories of permitting:

- Environmental Permitting: Confirmation that the applicable Environmental and Social Impact Assessment (ESIA) and its associated environmental management plan have been prepared, submitted, and approved in accordance with the requirements of the DRC Code de l'Environnement or other applicable approval, remains valid and in force as of the effective date of this Technical Report.
- Operational Authorizations: Confirmation that all requisite approvals from the Ministry of Mines, the provincial mining division (Division des Mines), and any other competent authority necessary to conduct activities currently being undertaken or proposed at the Project are in place, current, and in compliance with their respective terms and conditions.
- Water Use Permits: Confirmation of the currency and validity of any permits or authorizations required for the extraction, use, or discharge of water in connection with Project operations, as required under applicable DRC legislation.
- Land Access and Surface Rights: Confirmation that the Issuer holds valid and current surface rights or has secured lawful access agreements with landowners and / or occupants sufficient to conduct the activities described in this Technical Report, in accordance with the requirements of the Code Minier and applicable customary or statutory land tenure frameworks.

The QP notes that DRC regulatory and permitting frameworks are subject to ongoing administrative interpretation and may be affected by changes in government policy, personnel, or enforcement priorities. The QP has not independently assessed compliance with all applicable DRC regulations and has relied on the Issuer's representations that no material permit violations, suspensions, or regulatory enforcement actions are outstanding or pending as of the effective date of this Technical Report.

3.4 Responsibilities of the issuer

The Issuer has represented to the QP, in writing, that:

i. All information provided to the QP with respect to legal tenure, mineral rights, and permitting is accurate and complete in all material respects as of the effective date of this Technical Report.
ii. The Issuer is not aware of any facts or circumstances that would render any representation relating to tenure or permitting materially misleading.
iii. The Issuer accepts responsibility for the accuracy of the information disclosed in this chapter to the extent that such information is based on the Issuer's representations, legal advice received by the Issuer, and documentation provided to the QP.

The QP has no reason to believe that such representations are false or misleading but has not independently verified them.

3.5 QP statement of reliance

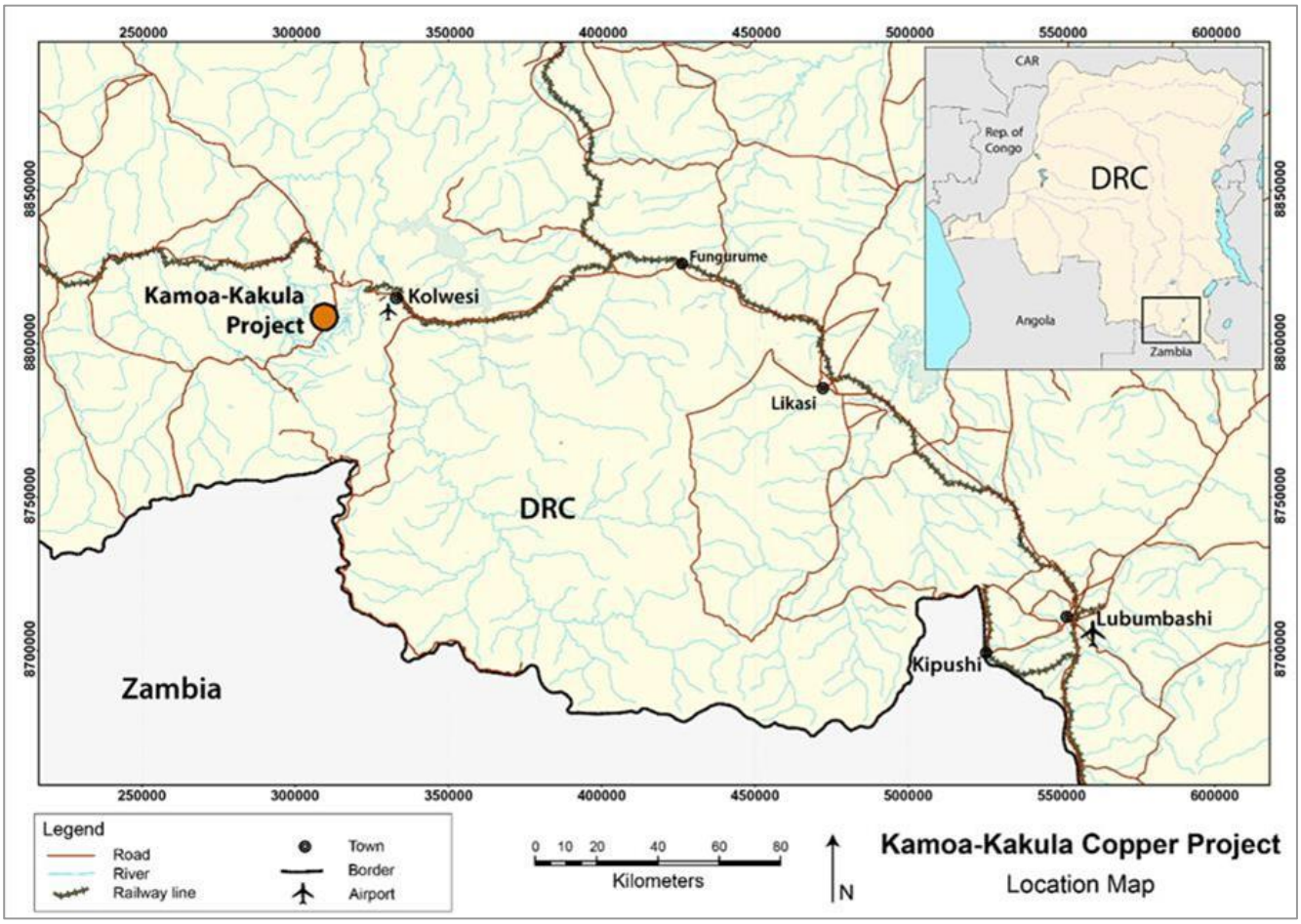
In accordance with Section 5.2 of NI 43-101, the QP states that, to the extent this Technical Report contains information relating to legal tenure and the currency of applicable permitting in the DRC, such information has been provided by the Issuer and is not within the independent expertise of the QP. The QP has taken reasonable steps to review the documentation provided but does not accept responsibility for the accuracy of such information beyond the exercise of reasonable professional diligence in the context of this Technical Report.

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 centered 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



Source: 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 summarized 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 recognized 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 favor 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 kilometers 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 nationalized 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 authorized 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 several 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--->-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--->-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 authorization 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 authorization 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 authorization and subsequently to a meeting in this respect with the Provincial Minister of Mines for Lualaba, KCSA is in the process of preparing a letter to the relevant authorities to

confirm, as soon as practically possible, that the Province Governor's authorization 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 authorized.

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 authorizations (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 authorizations applied for and paid for or to be paid for, when required by the relevant administrations.

4.2.5.1 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 minimized 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.

4.2.5.2 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 9 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

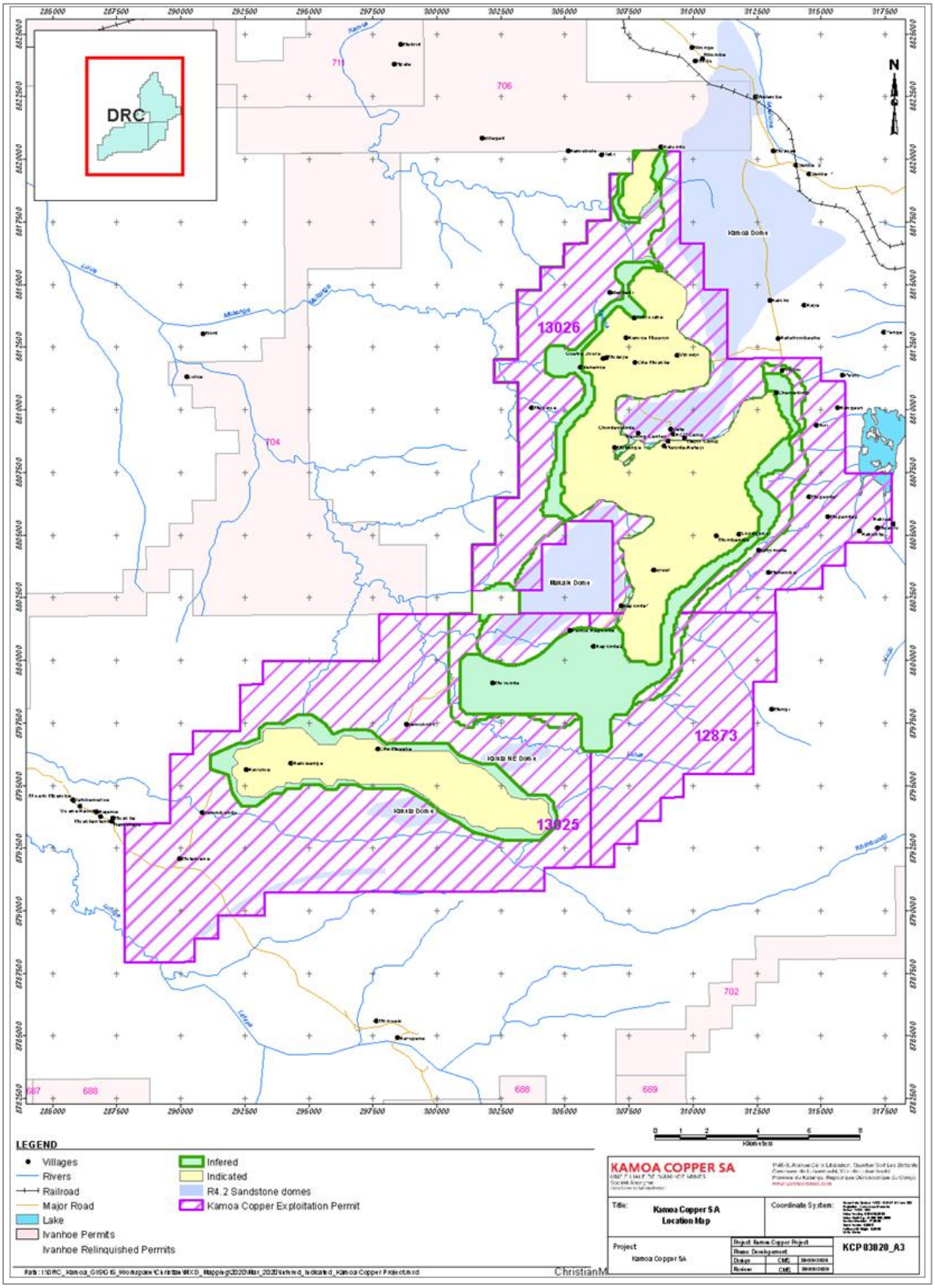
The Kamoa-Kakula Project consists of the Kamoa exploitation permits (Exploitation Permits No. 12873, 13025 and 13026 which cover an area of approximately 39,316 hectares). A mineral tenure summary table is provided in Table 4.1 and the mineral tenure locations are as indicated in Figure 4.2. The exploitation 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, Sulphur 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, Sulphur 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, Sulphur and Zinc.	202	16,972.25
Subtotal					**39,315.61**

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

Figure 4.2 Project tenure plan



Source: 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 authorized 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 programs 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 industrialized 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 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 minimized 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 extended 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. The upgrade of unit 5 (G25) was completed in November 2025 and the available capacity will increase to 125 MW in May 2027.

As of the effective date of this report, KCSA is busy with the installation of the following Solar PV and BESS systems by IPPs. The Solar / BESS plants are baseload specified and can therefore also serve as standby supply.

- Kamoa #1 – 30 MW baseload due for commissioning June 2026.
- Kamoa #2 - 30 MW baseload due for commissioning August 2026.
- The following systems are in the planning phase and will be installed at Kakula.
- Kakula #1 – 30 MW baseload planned for commissioning December 2026.
- Kakula #2 – 30 MW baseload planned for commissioning January 2028.

The following Solar/BESS systems are planned to supply power into the future Kamoa KCS #2 substation.

- Kamoa #3 – 30 MW baseload planned for commissioning December 2029.
- Kamoa #4 – 30 MW baseload planned for commissioning January 2029.

Further discussion on Power is completed in Section 18 of this document.

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 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, characterized 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 required infrastructure. The requirements to establish such infrastructure are reasonably well understood by KCSA. 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 mineralized 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 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 because 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 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.

The previous Technical Report was the Kamoa-Kakula Integrated Development Plan 2023 with an effective date in March 2023. This included the updates to the Project Mineral Reserves, and the studies at Pre-feasibility (PFS), and Preliminary Economic Assessment (PEA) stages.

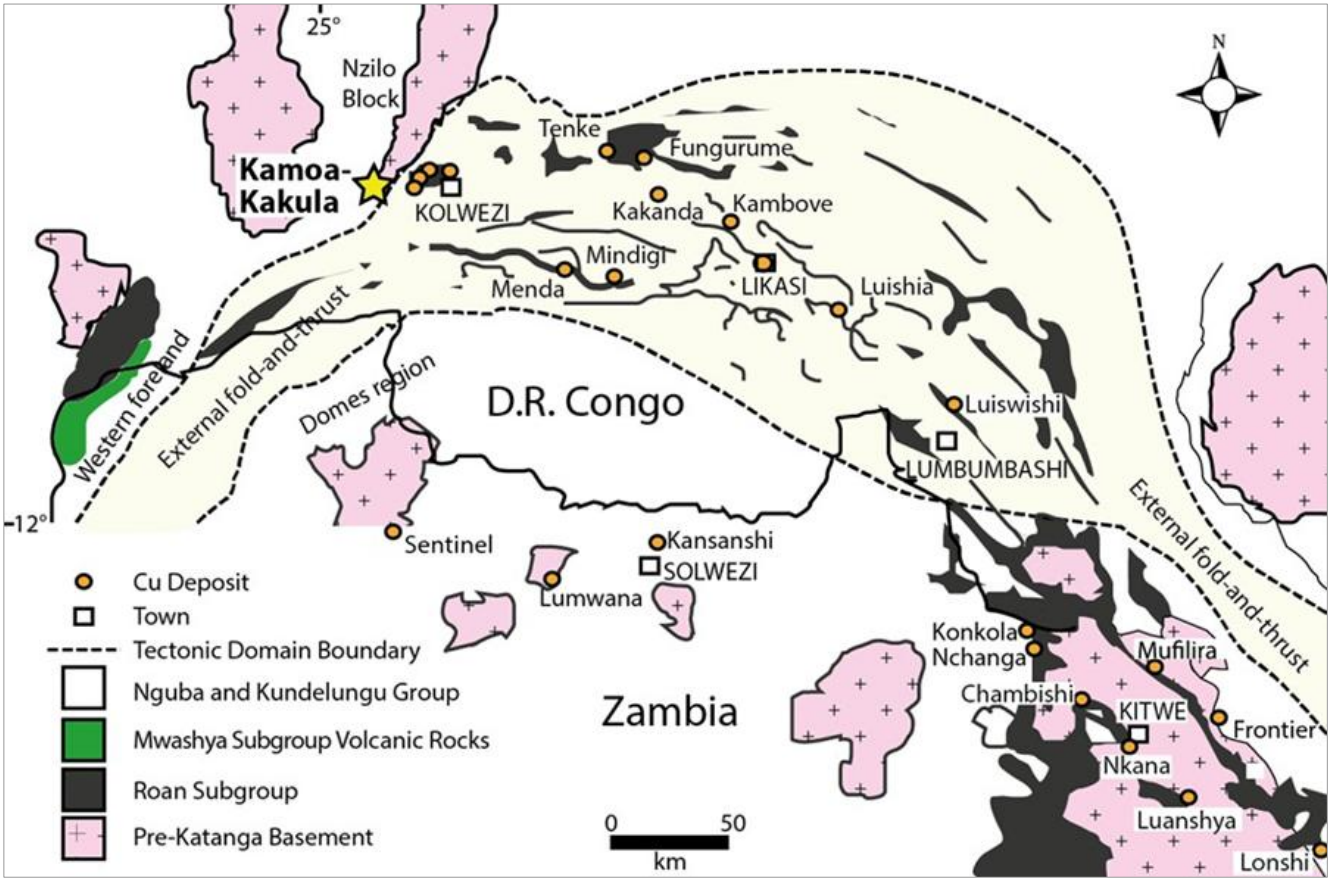
This current Technical Report supersedes all previously reported Mineral Reserve and Mineral Resource estimates for the KCSA Kamoa-Kakula deposits.

7 Geological setting and mineralization

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

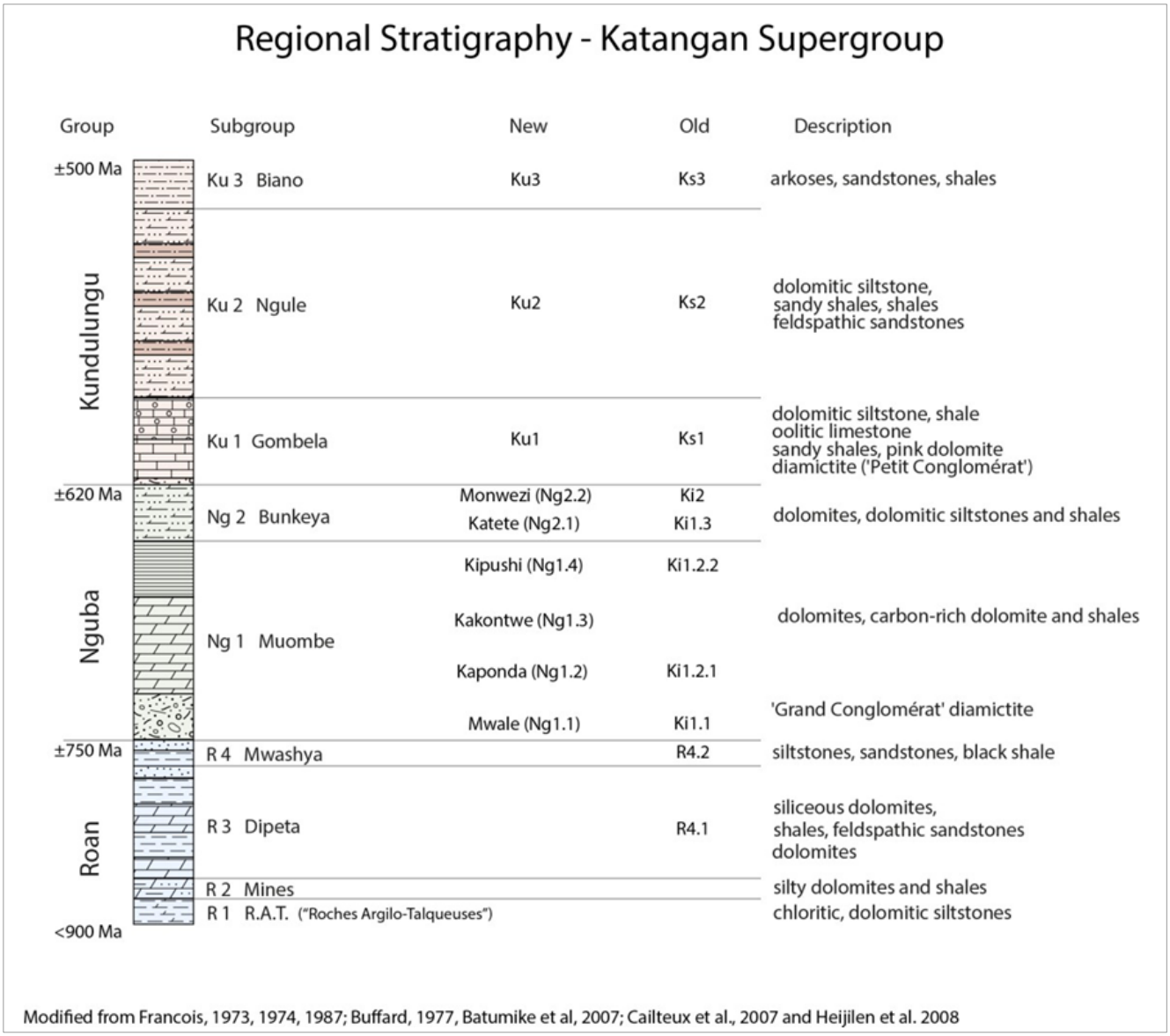


Source: Adapted from Schmandt et al (2013).

All of the Mines Subgroup copper (± cobalt) orebodies of the Katangan Copperbelt occur as mega fragments (écailles) up to kilometers 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 neighboring 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



7.2 Project geology

The modelled Kamoa deposit is located in a broadly folded terrane, with the antiform centered 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 (oxidized) Roan sandstones (Mwashya Subgroup), and the overlying carbonaceous and sulphidic (reduced) Grand Conglomérat diamictite (N'Guba Group), which comprises diamictites with minor interbedded sandstone, siltstone, and conglomerate. The mineralization 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 sulphide mineralization in the diamictites and siltstone units at the base of the Grand Conglomérat (Schmandt et al. 2013).

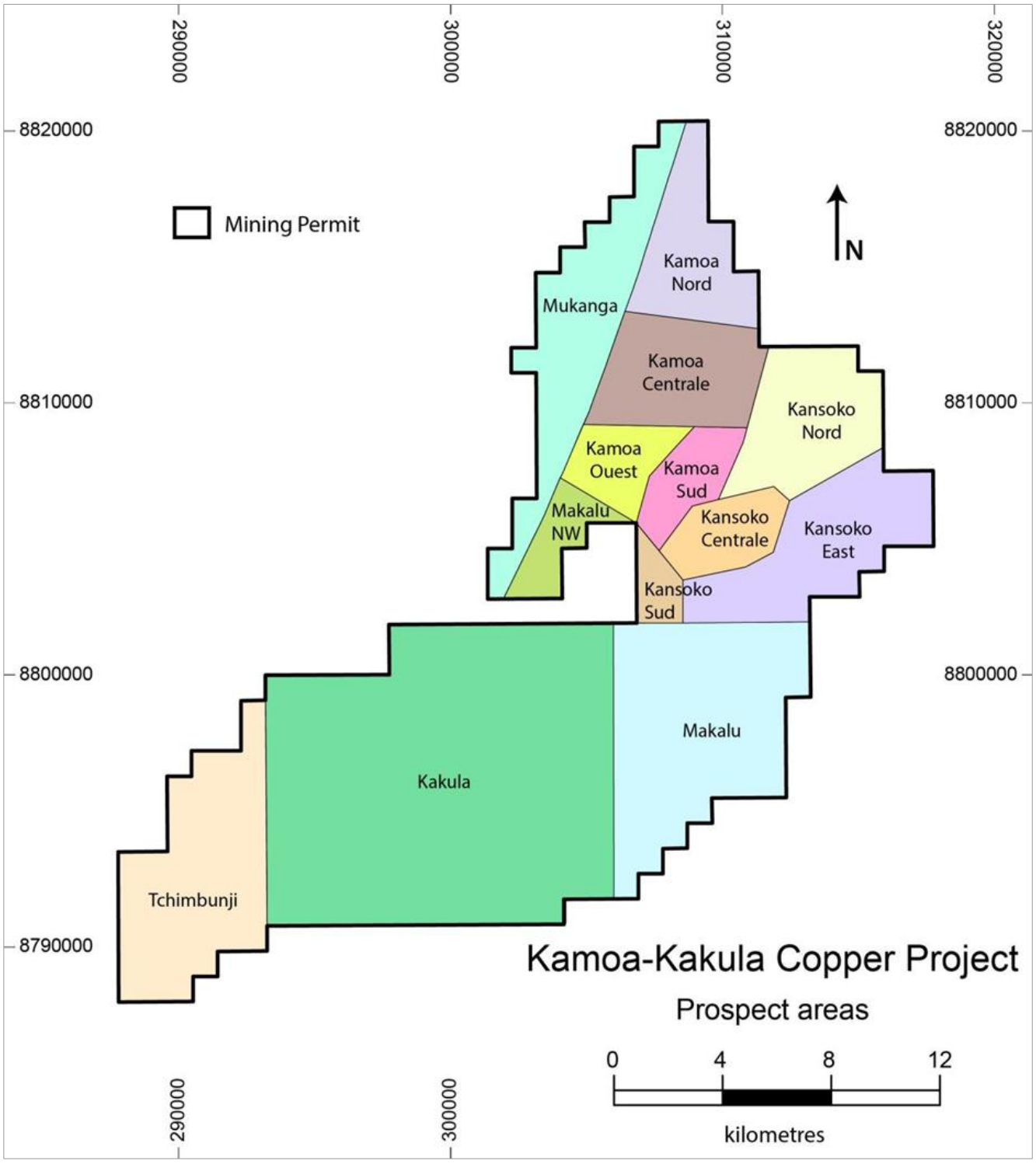
Andesite / dolerite sills occur as one or more, 5 – 80 m thick, apparently concordant tabular bodies in the extreme north-east of the Project area. The Katangan rocks in the Project area are weakly metamorphosed to lower greenschist facies. Alteration minerals include carbonate, chlorite, sericite, potassium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

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



Source: 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 oxidized 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 recognized within the basal diamictite, a clast-rich diamictite (Ng1.1.1.1), which is overlain by a clast-poor diamictite (Ng1.1.1.3). Mineralization 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, mineralization 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 mineralization 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.

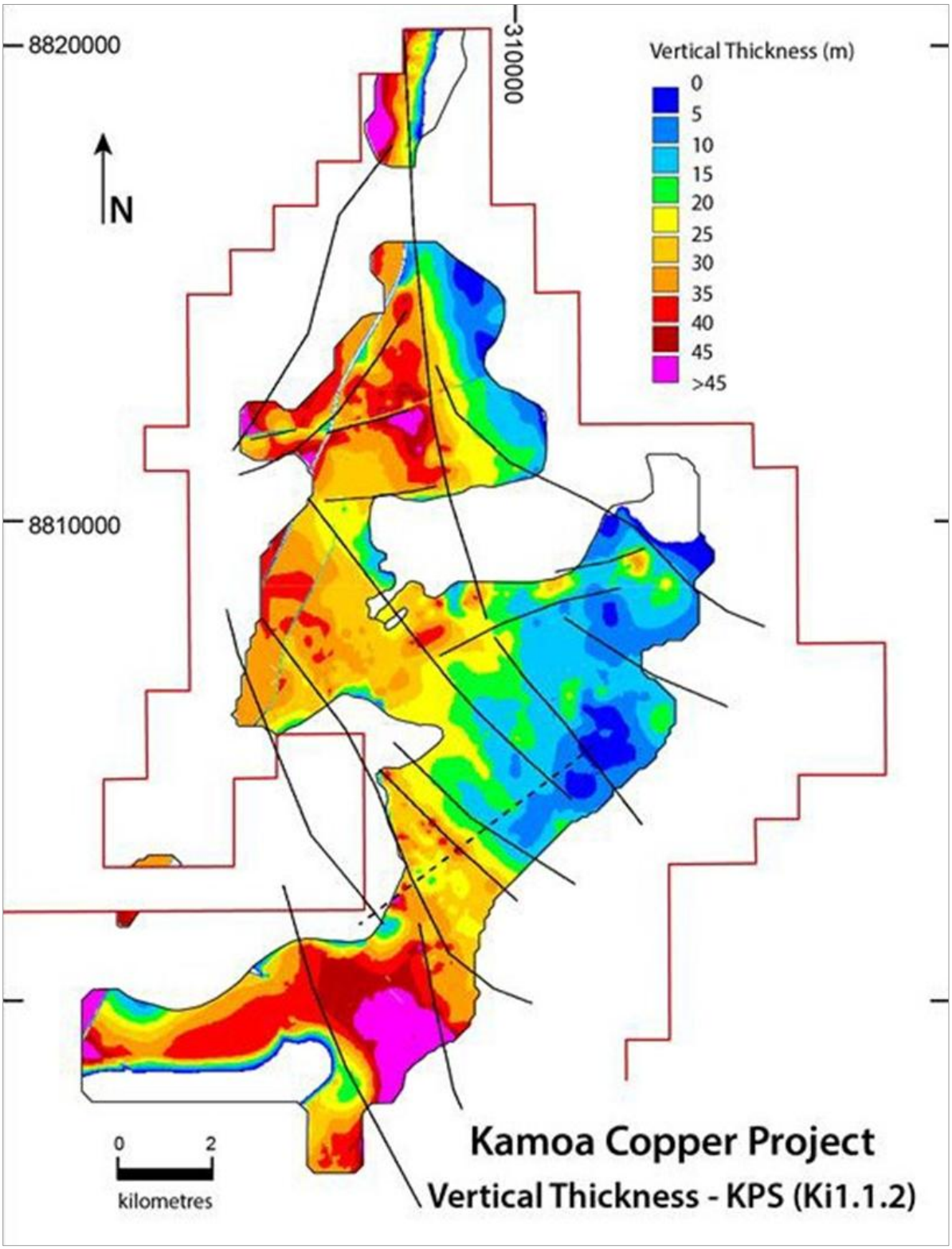
The stratigraphic units generally dip gently at 5–20° away from the Kamoa, and Makalu dome edges. The Kamoa, and Kamoa North areas, are particularly gently-dipping; Kansoko Sud, and Kansoko Centrale, generally dip at 10–20° to the south-east, with occasional steepening up to 30°. The steepest-dipping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

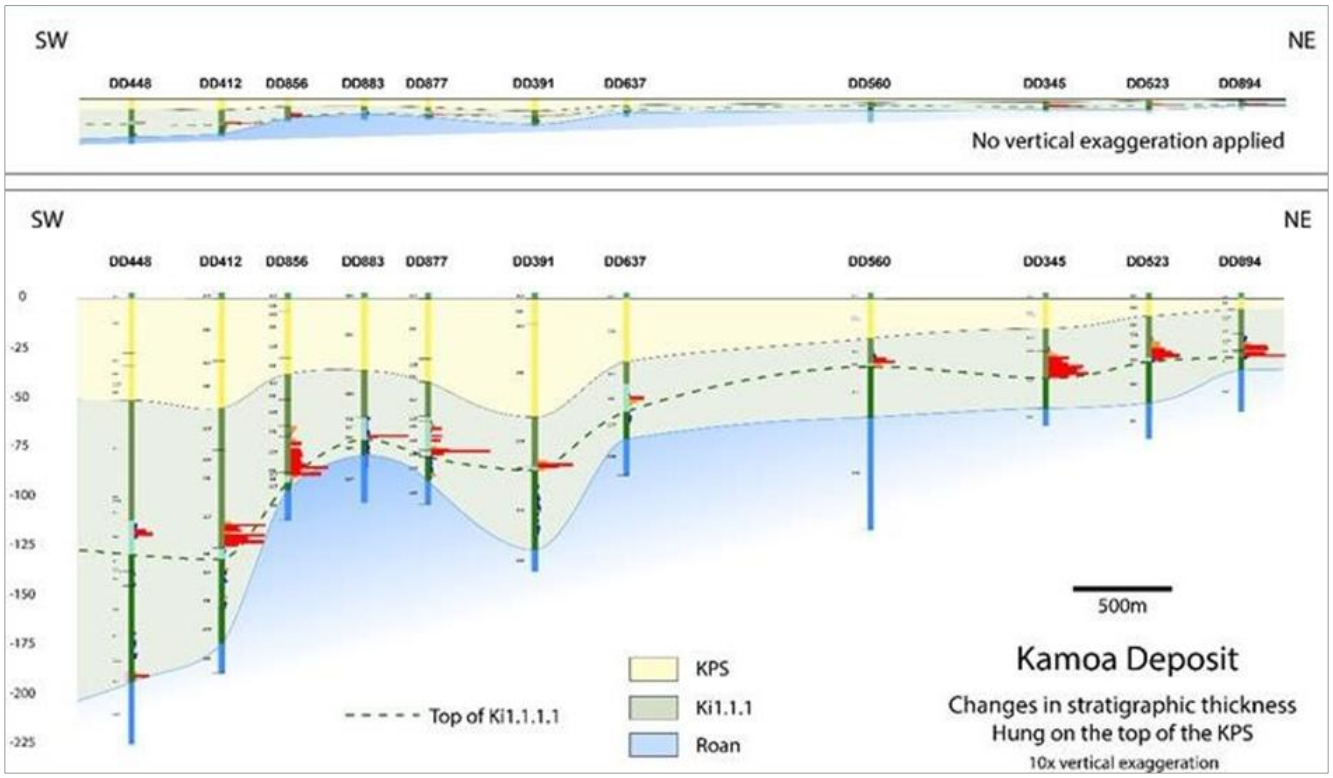
In the south-west, the thickening of the diamictite units is also marked by the development of thicker siltstone-sandstone-siltstone units, or the development of numerous siltstone units, comparable to the numerous siltstone units identified within the Ng1.1.1 at Kakula.

Figure 7.4 KPS (Ng1.1.2) vertical thickness



Source: Ivanhoe, 2020
Note: 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)



Source: Ivanhoe, 2014
Note: Illustrating the thickening of units to the south-west; section line location is indicated by dashed line. Copper grades are shown as histograms, with red being >1% TCu.

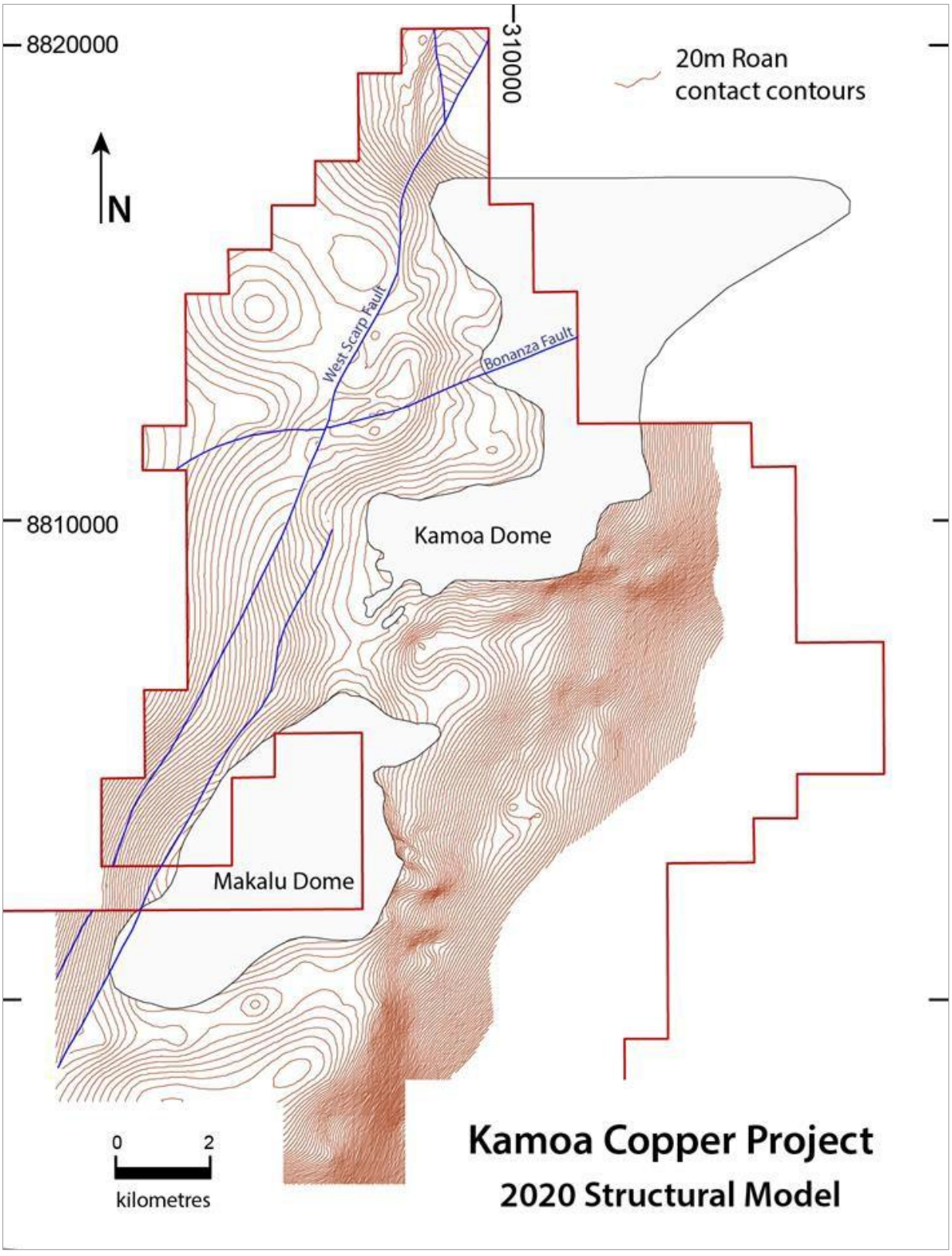
7.3.3 Structure

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 characterize more localized structures. Four major structures have been recognized, 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 mineralization into structural zones, refer to Figure 7.6.

The presence of very open folds at Kamoa is 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



Source: Ivanhoe, 2020

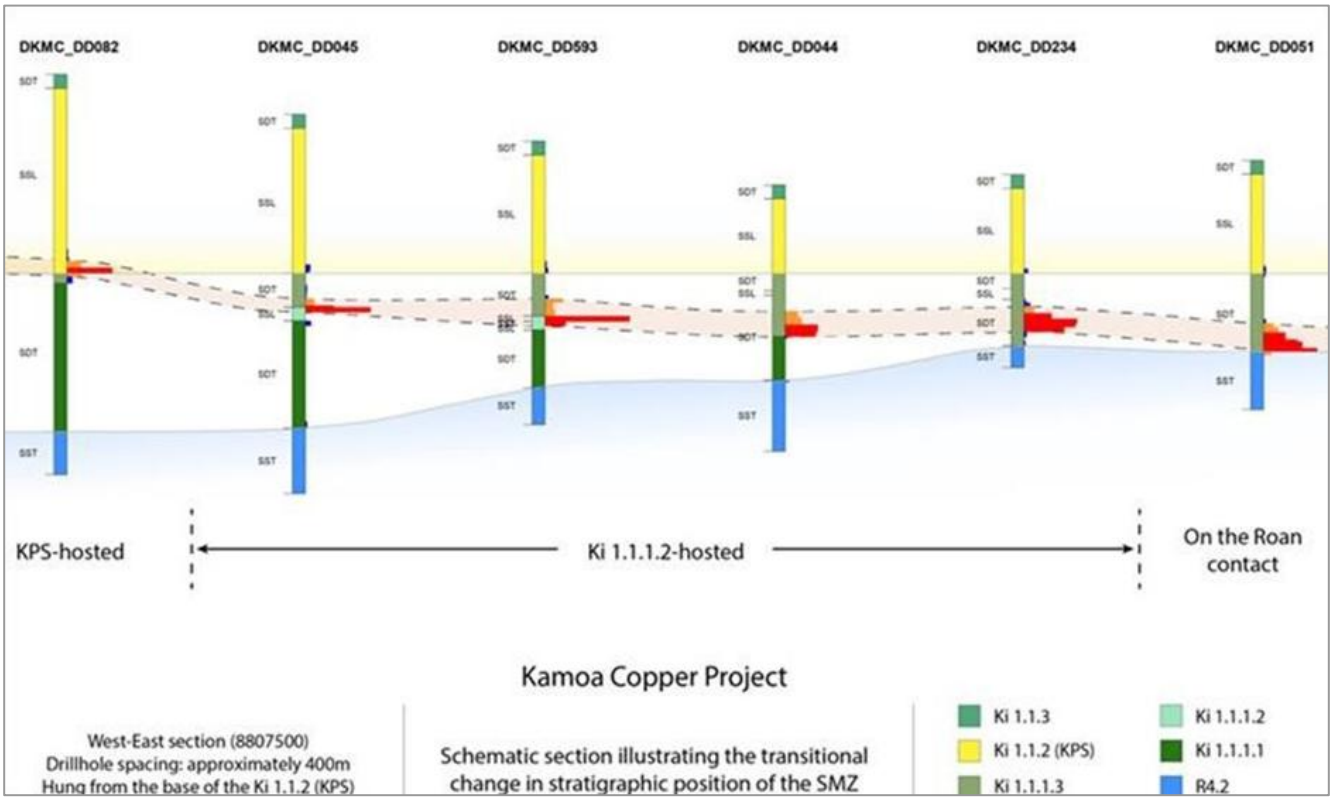
7.3.4 Mineralization

Mineralization at Kamoa has been defined over an irregularly-shaped area of 24 km x 14 km. Mineralization 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) mineralization. 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 mineralized 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 mineralization relates to the location of the reductant/s and proximity to the Roan aquifer. Although broadly stratiform, mineralization does transgress stratigraphy when a lower reductant narrows or pinches out. Mineralization 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 mineralization moves consistently and predictably from one unit to another (Figure 7.7).

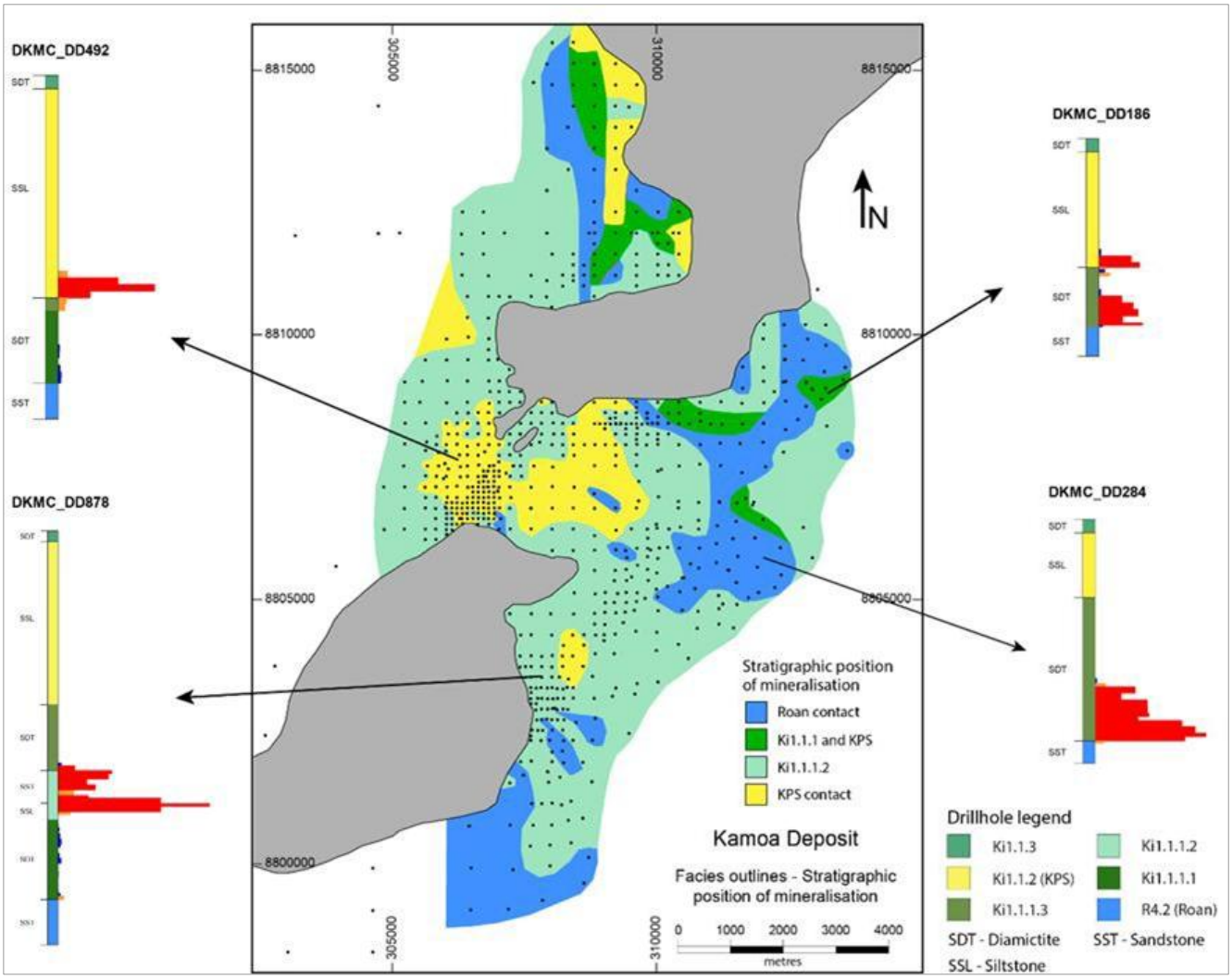
Figure 7.7 Stratigraphic section showing continuity of mineralization near base of Ng 1.1.1.3 at the Kamoa Deposit (8807500N looking north)



Source: Ivanhoe, 2014
Note: 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 localized development of lithological units. Where the mineralization is located on the Roan contact, the mineralized interval is thick and has a very strongly-developed bottom-loaded profile. Where the mineralization 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 mineralization 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 mineralization has been identified across the Project (Figure 7.8).

Figure 7.8 Facies in which mineralization occurs



Source: Ivanhoe, 2016
Note: 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. Favorable 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 sulphide 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 recognized 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).

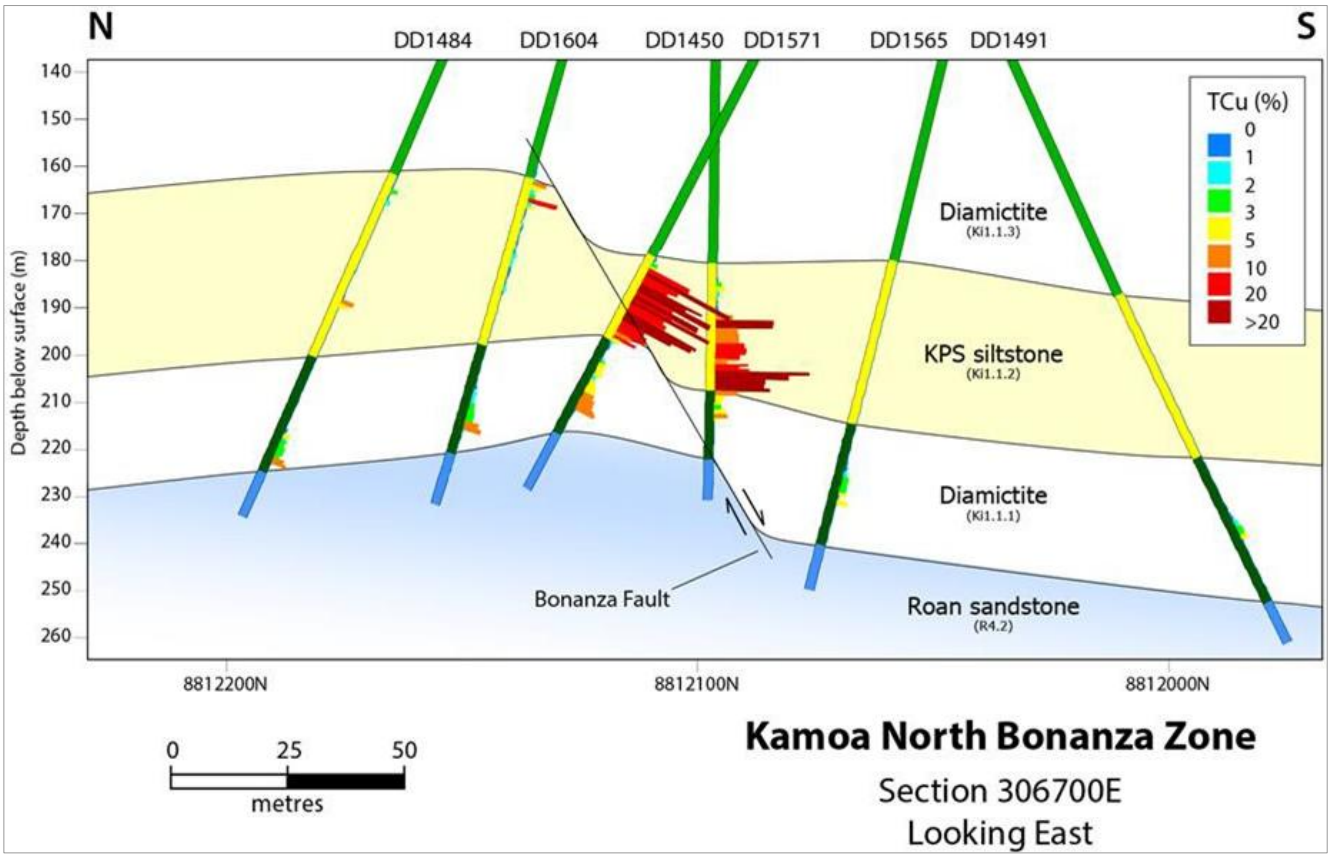
Supergene zones, in close proximity to dome edges, are typically fine-grained chalcocite dominant with secondary native copper and cuprite. The supergene zone may locally extend to depths of 250 m or more along fracture zones.

Since 2018, exploration has primarily focused on targets in the Kamoa North, and Kamoa Far North regions. Within the Kamoa North region, a new style of mineralization 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 focusing copper-rich fluids to interface with both the typical mineralized horizon at Kamoa, and the overlying, highly-sulphidic and reduced KPS (Ng1.1.2; refer to Figure 7.9). This has resulted in a stacked mineralized horizon, with the upper mineralized 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 mineralization 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 mineralization 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 mineralization along an approximately north–south trending fault, where fluids have been focused 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



Source: 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) recognized at Kamoa. A key lithological unit recognized 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 mineralization.

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.

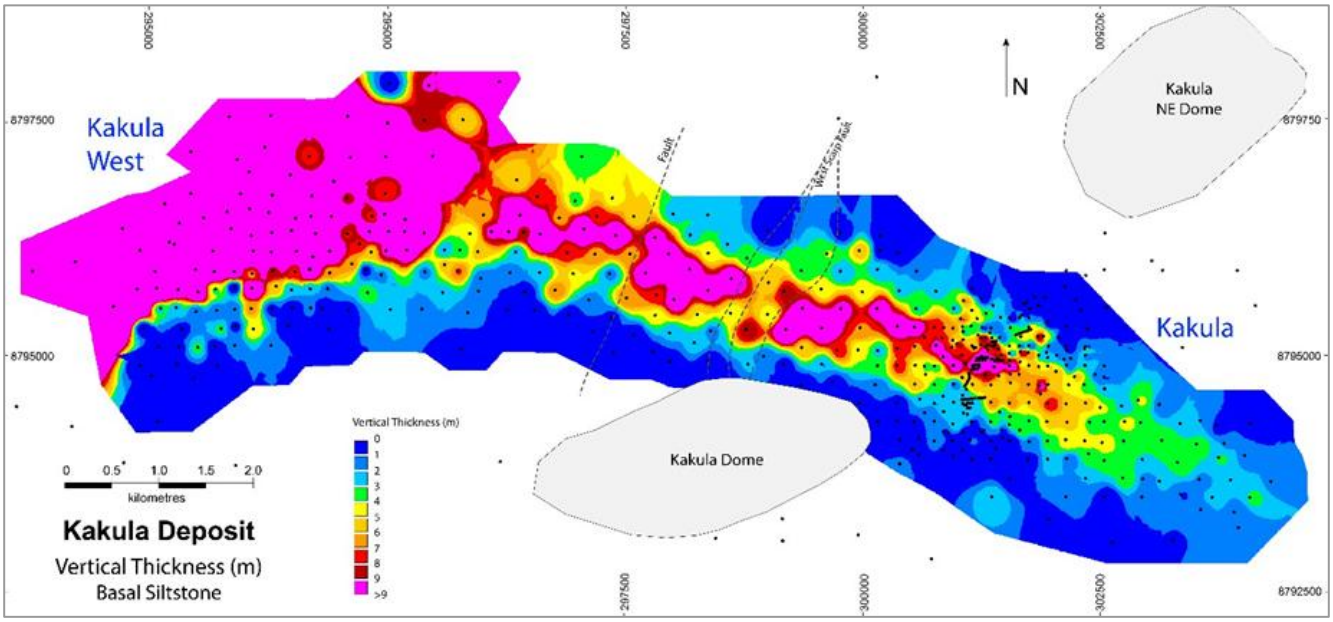
7.4.2 Thickness of modelled units

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 cannot 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.

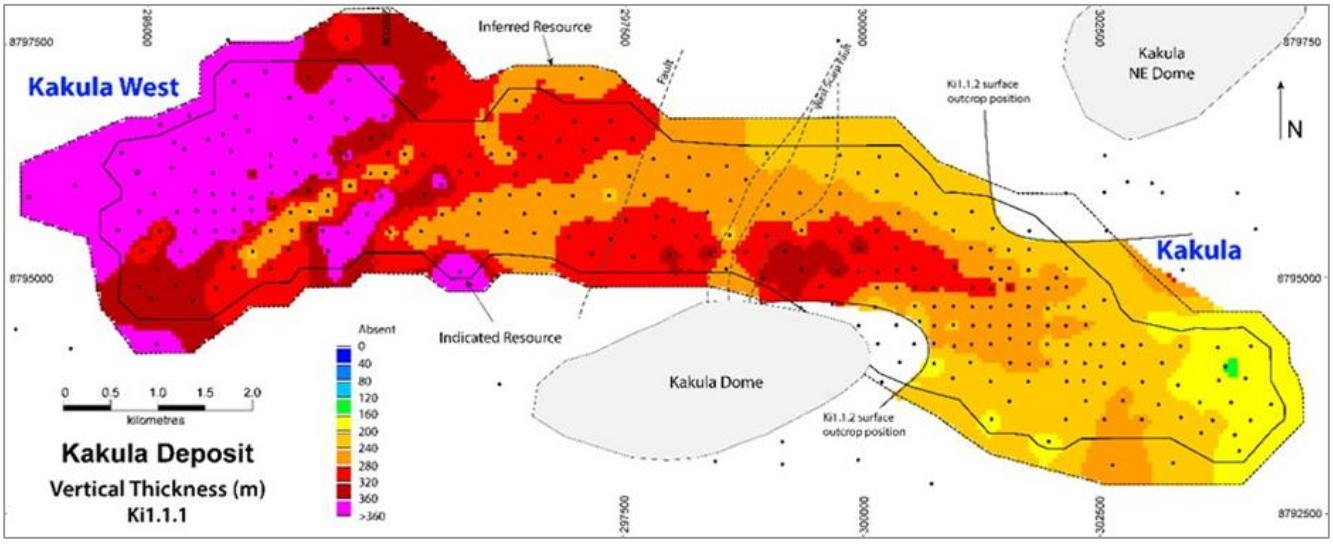
A pronounced east–north-east orientation in thickness trends is evident at Kakula West, and this observation has been incorporated into the search orientations used during grade estimation.

Figure 7.10 Vertical thickness of the basal siltstone within the NG1.1.1 at the Kakula Deposit



Source: Ivanhoe, 2023
Note: Vertical thickness estimated using an isotropic search.

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



Source: Ivanhoe, 2023
Note: Vertical thickness estimated using an isotropic search.

7.4.3 Structure

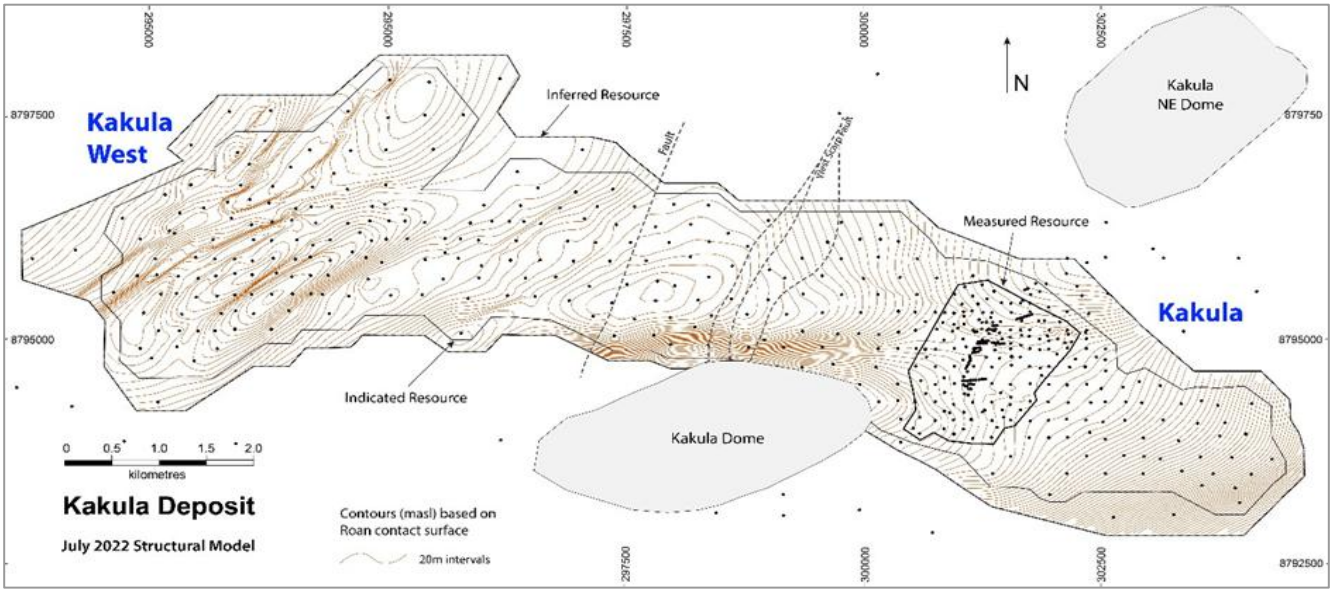
The geometry of the Kakula deposit is strongly influenced by extensional faults. Because the faults were active during deposition, a number of sub-basins were formed across the axis of a broad doubly-plunging antiform, and lithological units appear to drape across the extensional faults rather than having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

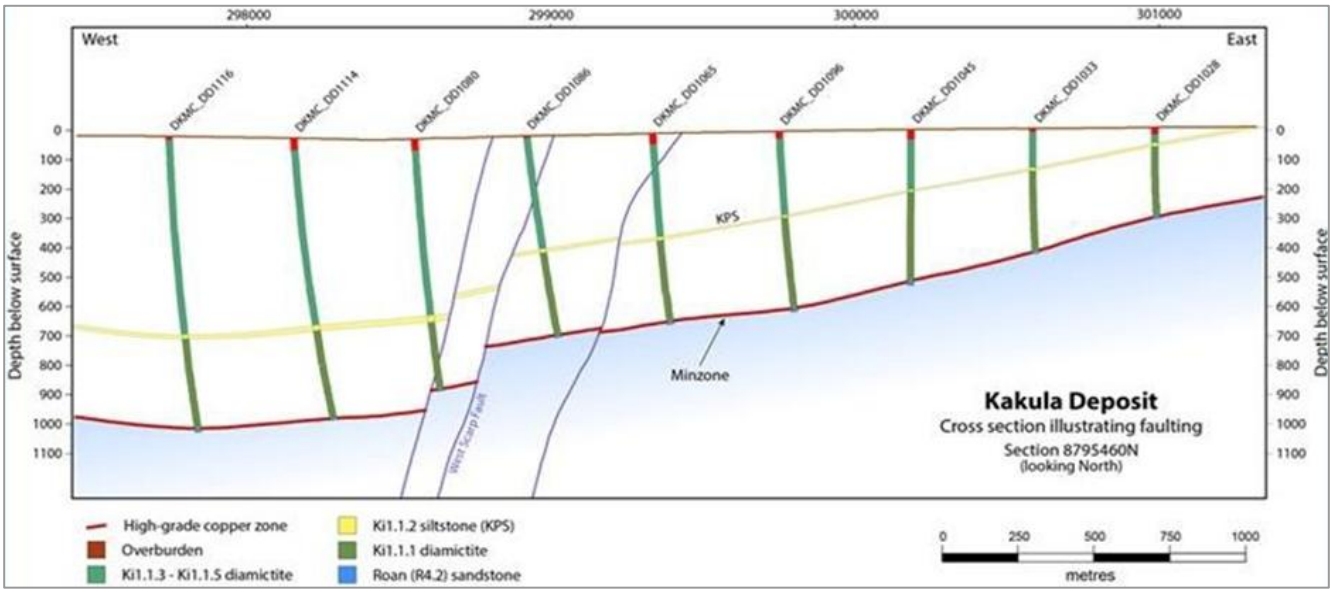
Younger brittle structures are also observed at Kakula that locally offset the mineralization (Figure 7.12 and Figure 7.13). The most prominent of these faults trend north–north-east and are related to the West Scarp Fault. Additional observed structures in drill core include steeply-dipping chaotic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



Source: Ivanhoe, 2023

Figure 7.13 Long section of the north-west Kakula area illustrating offset across the modelled faults



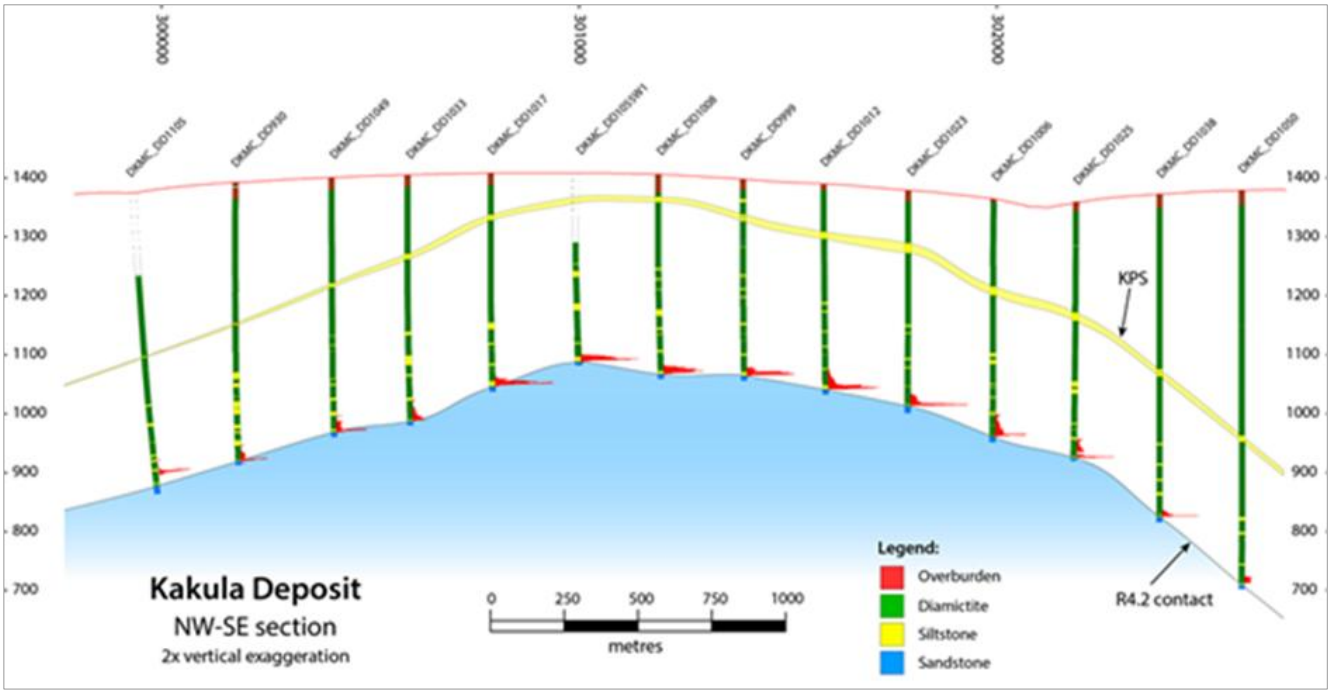
Source: Ivanhoe, 2018

7.4.4 Mineralization

The Kakula deposit is currently delineated over an area of 14 km by 5 km. The vertical thickness of the mineralization 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 mineralization relative to the basal siltstone (Figure 7.14). This relationship is considered to represent the distribution of the pyrite reductant prior to mineralization 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

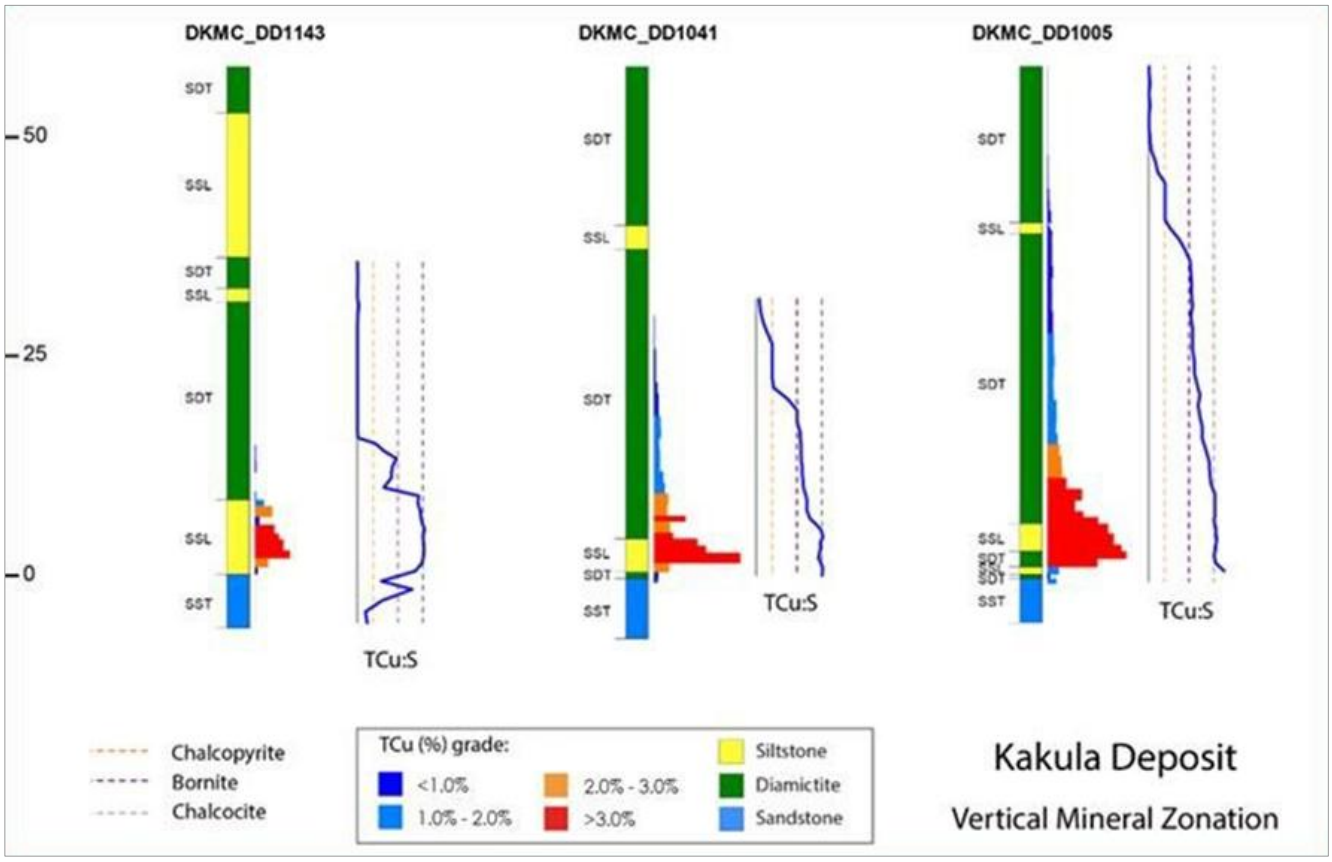


Source: Ivanhoe, 2017
Note: Red bars indicate assay intervals grading ≥ 0.5%.

Mineralization 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 mineralized zone, except in peripheral areas at Kakula West where the overall mineralized 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



Source: Ivanhoe, 2018

In the south-eastern portions of Kakula, the highest-grade intersections trend 115°, and align with the different stratigraphic and lithological units. To the north-west, the mineralization 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 QP notes the following:

- The understanding of the deposit settings, stratigraphy, lithology, structures, sulphide mineralogy, alteration, and their controls on the mineralization are well understood and sufficient to support estimation of Mineral Resources at Kamoa and Kakula.

8 Deposit types

The mineralization 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 oxidized 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 mineralized zone varies from sheet-like, with extensive horizontal dimensions, to tabular or roll-front geometries, with limited horizontal dimensions.

Mineralization:

- Deposits consist of relatively thin (generally <30 m and commonly less than 3 m) sulphide-bearing zones, typically consisting of haematite–chalcocite–bornite–chalcopyrite–pyrite. Some native copper is also present in zones of supergene enrichment. Galena and sphalerite may occur with chalcopyrite, or between the chalcopyrite, and pyrite zones. Minerals are finely disseminated, strata bound and locally stratiform. Framboidal or colloform pyrite is common. Copper minerals typically replace pyrite and cluster around carbonaceous clots or fragments.

Mineralization timing:

- Sulphides and associated non-sulphide minerals of the host rocks in all deposits display textures and fabrics indicating that all were precipitated after host rock deposition. Timing of mineralization 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.

Mineralization controls:

- Reducing low pH environment such as marine black shale; fossil wood, and algal mats are important as well as abundant biogenic sulphides and pyritic sediments. High permeability of footwall sediments is critical. Boundaries between hydrocarbon fluids or other reduced fluids and oxidized 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 mineralization 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 mineralization to a redox boundary zone between oxidized footwall haematitic sandstone and reduced, sulphidic host diamictites and siltstone-sandstone rocks.
- Presence of the replacement, blebby and matrix textures that are typical of sediment-hosted copper deposits.
- Vertical zoning of disseminated copper sulphide minerals from chalcocite to bornite to chalcopyrite.
- Hypogene minerals are chalcopyrite, bornite and chalcocite, with the predominant copper sulphide species varying spatially throughout the deposit. For example, deep drilling along the Kansoko Trend has intersected mixtures of bornite and chalcocite. Mineralization at Kakula is predominately chalcocite.
- Occurrence of very fine-grained, bedded, disseminated copper sulphides 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 mineralization.

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). Localized minor dolomite replacement of sulphidic clast rims in the basal diamictite and scattered tiny carbonate +/- quartz veinlets with occasional sulphides can occur at the Kamoa deposit.

Exploration 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 were used to create vectors into mineralization. Geochemical sampling programmes included stream sediment, soil, and termite mound sampling.

9.4 Geophysics

Geophysical surveys completed over the Project are summarized 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-mineralization faults, which appear to have large displacements.
Downhole electromagnetic (EM); Gap Geophysics Australia and Quik_Log Geophysics	2011	Orientation survey in three holes at Kamoa. Included natural gamma, density, sonic, magnetic susceptibility, three component magnetics, resistivity, conductivity, induced polarization and acoustic data (fractures).
EM orientation survey line	2011	Inconclusive results.
Ground magnetics	2011–2012	Used as a geology and structure mapping tool.
Ground gravity	2016	Eight lines at Kakula completed to help delineate the Ki1.1.1– R4.2 contact.
Downhole surveys; Quick Log Geophysics	2016–2017	12 drillholes. Logged full wave sonic, dual density, resistivity and gamma, collected acoustic televiewer (ATV) data.
2D seismic; HiSeis	2017–2018	Four regional scale lines completed to position the top of the Roan, interpret broad-scale basin architecture and locate both known and unknown growth and younger brittle structures.
Radiometrics (Excalibur); ground gravity and ground magnetics (Ivanhoe)	2019	Airborne radiometric surveys were completed over the planned Kakula tailings storage facility footprint. Ground gravity, ground magnetics and airborne radiometrics were conducted in the Kamoa North area to better understand the controls of the very-high-grade mineralization.
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 mineralized zone and footwall sandstone in drillhole DKMC_DD019. Results indicated possible K_2O enrichment commensurate with potassic (feldspar–sericite) alteration.

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

Two additional studies have been summarized in papers released in the journals Sedimentology (Kennedy et al., 2018) and African Journal of Earth Science (Twite et al., 2019). These studies highlighted the importance of syn-sedimentary growth faults, and their role in localizing high-grades (Twite et al., 2019), and the origin of the thick diamictite packages as subaqueous debris flows (rather than primary 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 mineralization and for new zones of mineralization within this same horizon. With more drilling, the exploration potential for expanding the area of known mineralization 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. Beyond these drillholes the mineralization 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 QP notes:

- The exploration programmes completed to date, are appropriate to the style of the Kamoa and Kakula deposits.
- The Project area remains prospective for additional discoveries of base-metal mineralization within diamictites around known dome complexes.

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. Data from drillholes of these types are not used for resource estimation. Core holes have been used for geological modelling, and those occurring within the mining lease and in areas of mineralization (drillholes on the Kamoa, Makalu and Kakula domes are excluded) have been used for resource estimation.

As of 2 December 2025, there were 2,898 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, unmineralized holes in the dome areas, unsampled underground cover holes, metallurgical holes, 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).

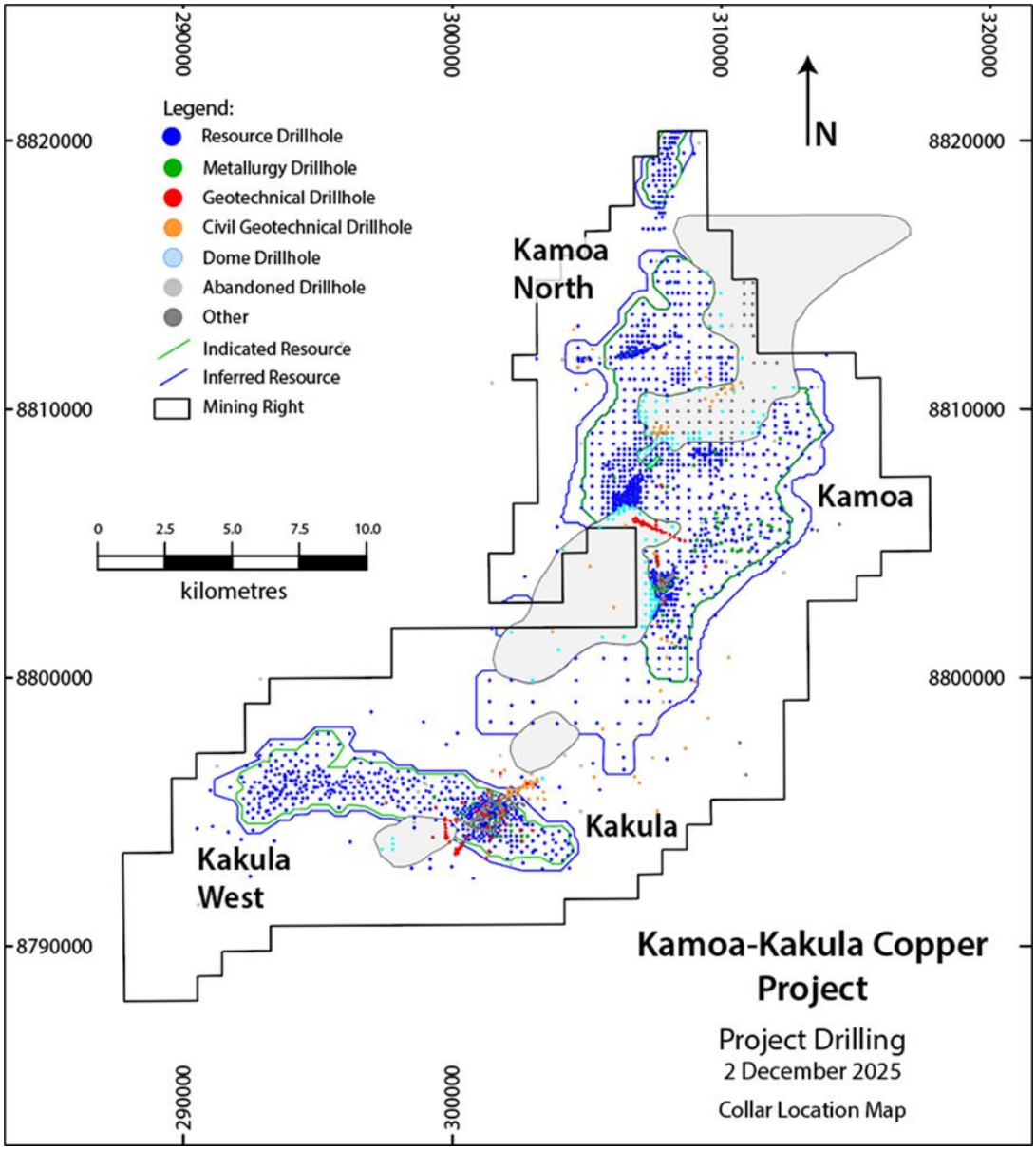
Table 10.1 Drilling statistics per drill purpose for core holes (as of 2 December 2025)

Drill Purpose	Count (Active)	Metres (m)
Kamoa estimate (Jan 2020)	998	288,140.7
Kamoa (post-estimation)	164	48,579.0
Kakula estimate (Dec 2022)	645	246,799.5
Kakula (post-estimation)	107	59, 887.1
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

Drill Purpose	Count (Active)	Metres (m)
Abandoned	131	28,892.5
Total	2,898	760,614.8

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



Note: 'Other' includes exploration drillholes, condemnation drillholes, cover drillholes, and hydrology drillholes.
Source: Ivanhoe, 2025.

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 chips, and aircore chips were logged by a geologist using paper forms, which capture lithological, weathering, alteration, mineralization, structural, and geotechnical information. Logged data were then entered into Excel spreadsheets using single data entry methods. Since 2012, all logging data have been captured electronically in the core yard using acQuire software, 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 analyzed to provide an initial estimate of the amount of copper present in the drill core. The results of the Niton XRF are not used for Mineral Resource estimation.

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 was 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 mineralized 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 mineralized 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 finalization 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 Downhole surveys

10.5.1 Kamoa

Core hole orientations ranged from azimuths of 0° to 360°, with downhole inclinations that ranged from -5.0° to vertical. Most holes were vertical or subvertical, with only the geotechnical drillholes (-45°) and cover drillholes (<-10°) at the Kansoko Sud and Kakula declines being shallow. Downhole surveys for most drillholes were performed by the drilling contractor at approximately 30 m intervals for 2009 drilling and at a maximum interval of 50 m for 2010 through 2020 drillholes using a Single Shot digital downhole instrument. Once the hole was completed, a Reflex Multi Shot survey instrument was used to re-survey the hole to confirm the Single Shot readings.

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

10.5.2 Kakula

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

10.6 Geotechnical drilling

Ivanhoe collects geotechnical and structural information from resource drillholes, 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 16.2.

10.7 Metallurgical drilling

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

10.8 Drilling since the Mineral Resource database close-off date

10.8.1 Kamoa

The database contains 164 drillholes (48,579.0 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.

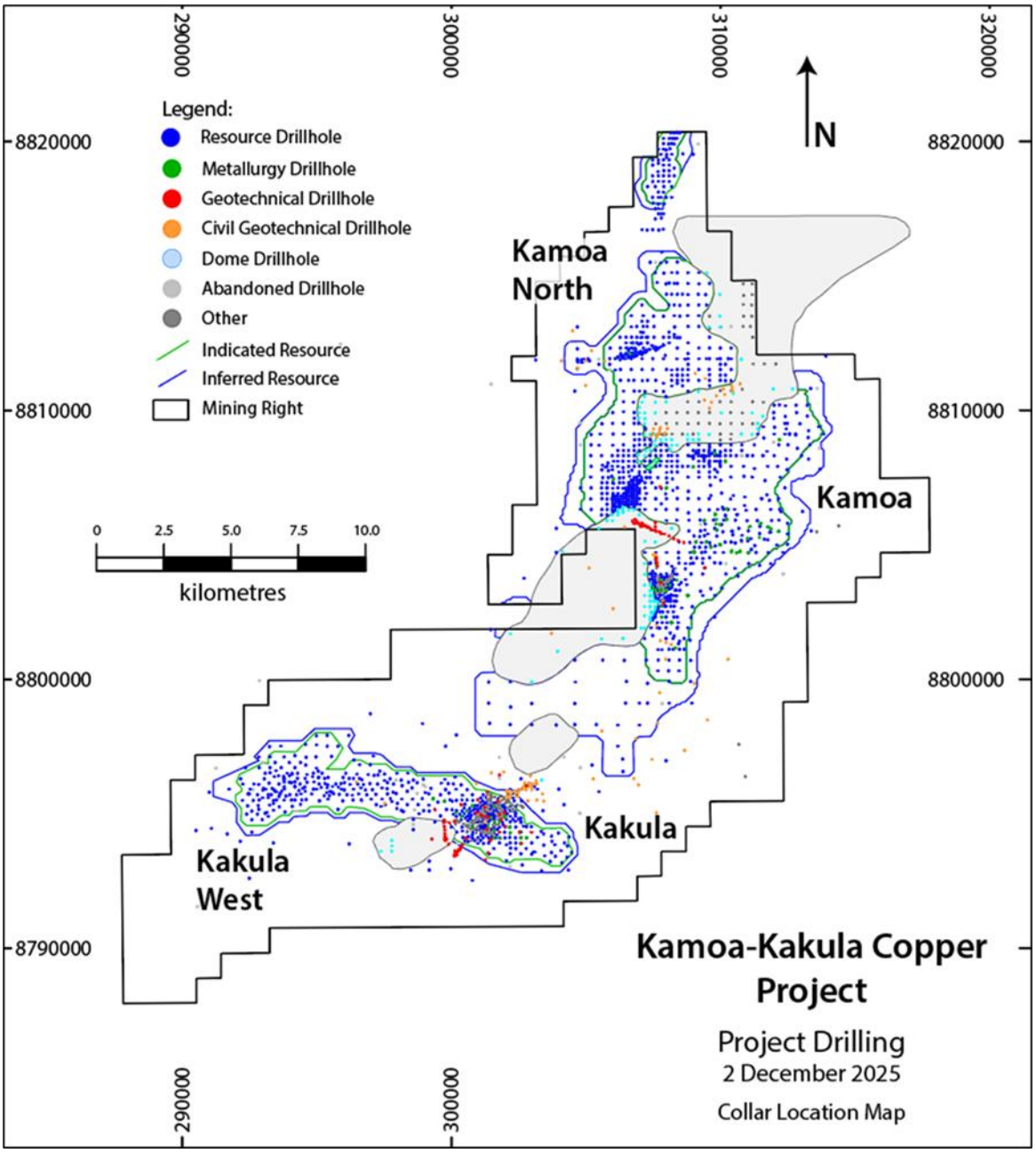
Although a few of the newer drillholes are very high-grade and may change the grades locally, the majority of the holes are within the existing model, and the MSA QP considers that the new drilling should have no material effect on the overall tonnages and average grade of the current Mineral Resource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

10.8.2 Kakula

Since 20 July 2022, Ivanhoe completed an additional 107 core drillholes (59,887.1 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 (as of 2 December 2025)



Source: Ivanhoe, 2025.

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 programs is sufficient to support Mineral Resource estimation. 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.
- Drilling has been completed since the database closure dates for the Mineral Resource estimate. While the results of the more recent drilling are not likely to impact on the grade and tonnage of the global estimates, they could enhance the accuracy of local estimates and potentially upgrade Mineral Resource classification. It is recommended that Ivanhoe update the Kamoa and Kakula Mineral Resource estimates periodically as new information is gathered.

11 Sample preparation, analyses, and security

11.1 Witness sampling

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

11.2 Sampling methods

11.2.1 Geochemical sampling

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

- 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 sampling 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 homogenization with the next metre sample in order to create a 2 m composite sample.

11.2.3 Core sampling

The core sampling procedure is as follows:

- Sampling positions for un-oxidized 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 mineralization, and / or the mineralized 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 mineralized basal diamictite was conducted as follows:

- The mineralized 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 mineralization in each drillhole.
- After March 2011, 9 m composite samples were collected in the hanging wall, and the prepared pulp was analyzed by Niton. The results are used to characterize the geochemistry of the hanging wall material.

- After August 2014, whole core was logged by the geologist on major lithological intervals until mineralized 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 mineralized 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 mineralized or material that was expected to be mineralized. This “zone of assay” (ZA) was extended to 3 m above and below the first sign of visible mineralization.
- 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 downhole, with the half section that contains the core orientation line retained in the core trays for geological logging and record purposes. The half-core along the right-hand side of the projected orientation lines was 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) was pulverized 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 test work.

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 optimization using Kakula material. PQ holes are used for comminution test work, while either HQ and / or NQ wedges are used for flotation test work programs.

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 summarizes 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 was located on the Kamoa-Kakula site and was used for all sample preparation. The laboratory was 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 pulverizers, 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 pulverized to >90% -passing 75 µm, using the LM2 puck and bowl pulverizers. A 100 g split was 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 had a 2 mm screen test performed and a further 5% at the pulverization stage were checked using a 75 µm screen test. Pulp bags of the pulverized material were 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 sulphuric 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 analyzed 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 analyzed 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 analyzed for acid-soluble copper (ASCu) using a 5% sulphuric acid leach method at room temperature for 60 minutes; only 249 of the 6,640 samples obtained in 2008 and 2009 were submitted for ASCu analysis. The sampling prior to 2010 was mainly in the Kamoa area. 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 analyzed 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 mineralized basal diamictite were analyzed 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 analyzed samples for Cu, Fe and S (BVM method ICP102 - using four-acid digestion followed by ICP-OES) and for Ag and As (BVM method ICP302 -– four-acid digestion with ICP-MS).

ASCu analysis was performed on early drillholes by a 5% sulphuric acid cold leach followed by ICP-OES. ASCu analysis has subsequently been discontinued by Ivanhoe. At Kakula, no ASCu results exist for drillholes DKMC_DD1024, DKMC_DD1025, DKMC_DD1031, and DKMC_DD1033 onward.

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

11.8 Quality assurance and quality control

Quality assurance and quality control (QA/QC) samples were 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 were 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 pulverizing 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 at a level that were cause for 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 pulverized. The current procedure is for blank samples to be placed after visually-observed higher-grade mineralization.

11.8.1.2 Kakula

Blank2010 and BLANK2014 were used as coarse blanks for the Kakula drill programme. One blank per 20 samples was inserted prior to the samples being pulverized. 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 mineralization. Due to higher-grade mineralization 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 mineralization.

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 mineralization grades. CRMs were placed within mineralization 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 mineralized zones 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 programs 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 downhole survey data were entered at the Kamoa (site) office, and assay data were imported directly from electronic files provided by the assay laboratory.

Where they exist from older drill programs, paper records for all assay and QA/QC data, geological logging and specific gravity information, and downhole and collar coordinate surveys are stored in fireproof cabinets in Ivanhoe's Kamoa site office. All paper records are filed by drillhole for quick location and retrieval of any information desired. In addition, sample preparation and laboratory assay protocols from the laboratories are monitored and kept on file. Digital data are regularly backed up in compliance with internal company control procedures. The 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 on-site 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 cores have been moved from Kolwezi to the facility at the Kamoa site camp.

11.12 Comments on Section 11

In the opinion of the 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 mineralization changes in the core, are typical of sample intervals used for Copperbelt style mineralization in the industry.
- Samples are taken for assay depending on location, stratigraphic position and observation of copper mineralization.
- Sampling is considered to be representative of the true thicknesses of mineralization. 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 mineralization 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, downhole and collar survey data, geological and mineralization logging, assay, and specific gravity (SG) data. These reviews were documented and no significant errors were noted that could affect Mineral Resource estimation.

Since 2020, the 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 were also reviewed and compared to the Mineral Resource model.

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 were 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 of 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 programs 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 on Section 12

The MSA QP considers that the data verification programs 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 undertaken by various parties, focusing on the metallurgical characterization and flowsheet development for the processing of hypogene and supergene copper ores. These investigations culminated in the development of the IFS4a flowsheet, which supported the Kamoa PFS completed in March 2016.

In 2016, Kamoa Copper SA discovered the Kakula deposit, which is characterized by significantly higher copper head grades compared to the Kamoa deposit. Consequently, the Kakula project was fast-tracked. Metallurgical testwork on the Kakula deposit commenced in 2016, with subsequent programs supporting flowsheet development and optimization.

The Kakula Phase 1 and Phase 2 concentrators were successfully commissioned in 2021 and 2022, respectively, and have since ramped up to design throughput. Both Kakula Phase 1 and Phase 2 circuits are currently undergoing modifications aimed at improving metallurgical recovery, with commissioning of the upgraded circuits scheduled for completion by the end of Q2 2026. Testwork for the recovery optimization programme was initiated at the site metallurgical laboratory and subsequently advanced by XPS and Zijin Laboratories.

The Kamoa complex has also undergone expansion through its phased development plan. The Phase 3 concentrator, with a nameplate capacity of 5 Mtpa, was commissioned in June 2024 and has since ramped up, currently processing in excess of 30% of its design capacity. This development was supported by additional metallurgical testwork at XPS, including flowsheet optimization studies and a flotation variability testwork programme for material sourced from the Kansoko and Kamoa mining areas.

For the purpose of the Kamoa-Kakula MRMR Update Technical Report, the metallurgical test work conducted on the Kamoa deposit has been referenced in detail as background to the IFS4a flow sheet development process. A summary of the metallurgical test work conducted on Kakula material for the Kakula Phase 1 PFS and feasibility study is provided.

13.1.1 Metallurgical test work on the Kamoa Resource

A series of test work 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%.

This series of metallurgical test work programmes, defined as Phases 1-6, were completed on Kamoa drill core sample. 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. Phases 1-5 focused on metallurgical characterization 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 mineralization 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 XPS

Laboratories. The Phase 6 samples best represent ores to be processed according to the early years of the Kansoko PFS mine schedule. Further metallurgical testwork programs were subsequently conducted by Kamoa Copper SA in collaboration with XPS in 2023. These included flotation variability testwork campaigns aimed at further characterizing material from key mining areas, including Kansoko, Kamoa 1, and Kamoa 2, as feed sources for the Kamoa Phase 3 concentrator. It is noted that many of the Phase 2 and Phase 3 samples are relevant to the current Kamoa PFS mine schedule.

13.1.2 Preliminary metallurgical test work on Kakula Resource

The initial mineralogical and flotation test work on the Kakula resource was conducted during 20162017, 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 sulphide mineral in the Kakula samples was chalcocite, with minor amounts of bornite and covellite. Trace amounts of chalcopyrite were 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 test work 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 test work 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 test work campaign, which focused on flowsheet optimization as part of the Kakula PFS. Testwork completed during 2017–2018 included various mineralogical studies, comminution parameter testing, flotation flowsheet optimization, HPGR test work, concentrate and tailings thickening and filtration testing, bulk material flow test work, comminution variability test work, as well as flotation variability test work.

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

During 2017, and 2018, Mintek performed comminution characterization test work as well as preliminary variability test work 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. Crushing Work Index (CWi) values ranged from 9.8–13.5 kWh/t, characterizing 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 Bond Rod Mill Work Index (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 Bond Ball Mill Work Index (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 test work 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 was 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 material handling designs.

XPS conducted work on the Kakula material, to further optimize the IFS4a flowsheet, 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 flowsheet, assessment of primary grind, and optimization 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 recoveries. Moving 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 flowsheet 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 flowsheet. However, an improvement in the Cu and SiO_2 grades was made.

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

Following the Kakula PFS test work campaign, XPS conducted flotation variability test work 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 sulphur grades generally increasing with increasing Cu grades. Fe, MgO, and Al_2O_3 values were relatively constant over the range of samples, averaging 5.0%, 4.0% and 13.5% respectively. The highest arsenic value measured was 0.003% with the majority of the samples reported as below the instrument detection limit of 0.001%.

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

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

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

Overall, the samples tested across the Kakula deposit performed relatively consistently, on the Kakula flow sheet. The Cu mineralogy is variable and ratios between chalcocite, bornite, chalcopyrite and nonsulphide 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 test work on Kakula Resource

Following the completion of the Pre-feasibility Study, further test work was initiated in March 2019 as part of the Feasibility Study, and consisted of:

- A mini-pilot plant campaign including Jameson Cell test work, conducted by XPS.
- Desliming cyclone test work, conducted by Multotec, South Africa.
- Flocculant screening test work, conducted by ChemQuest, South Africa.
- Various slimes and full tailings settling test work, conducted by Outotec, Paterson & Cooke, and Andritz.
- Concentrate regrind hydro cyclone and signature plot test work, conducted by Grinding Solutions.
- Flotation tests utilizing 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 sulphide 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 test work reported an energy requirement of 20.14 kWh/t for the regrind step to achieve a combined product of P80 10 µm.

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

13.1.6 Metallurgical test work on Kakula West material

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

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

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.

13.1.8 Kakula Optimization Test Work Programme

Test work conducted by XPS, Zijin in China, and the Kamoa site laboratory was aimed at improving recovery from the Kakula circuit and reducing copper losses to tailings. Optimum performance was achieved through the implementation of several key modifications, including:

- Incorporation of a flash flotation circuit in the secondary milling stage to optimise recovery of coarse liberated chalcocite and minimise overgrinding
- Introduction of a split flotation circuit for coarse and fine fractions, with the coarse tailings subjected to an increased-capacity regrind circuit to improve liberation and overall recovery
- Reconfiguration of the cleaning circuit post-regrind to improve upgrade ratios while maintaining a high concentrate grade.

The test work flowsheet is illustrated in Figure 13.1. Three samples were collected for the programme. The primary grind target was 80% passing 53 µm, while the regrind target was 80% passing 10 µm. Sample 1 was used as the main composite sample, while Samples 2 and 3 were utilized for open-circuit variability test work. The outcomes of the test work are summarized below.

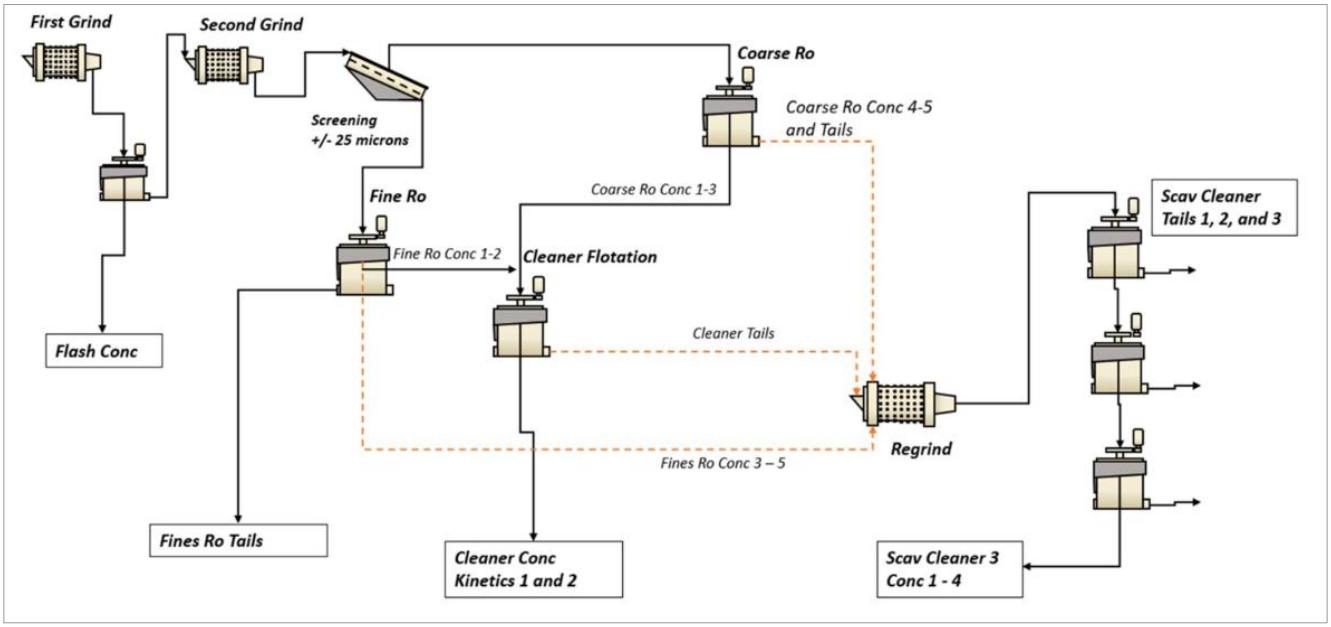
- Sample 1 grading 4.9% Cu in feed, achieved an open circuit copper recovery of 92.6% with a concentrate grading 55% Cu.
- Locked cycle test work with flash flotation and split coarse / fine rougher flotation flowsheet produced final copper recovery of 95.1% at 50.5% Cu concentrate grade.

Open circuit cleaner test work results for sample 2 and sample 3 are summarized below.

- Sample 2 flash flotation and split rougher flotation open circuit recovery was 91-92% at 46-50% Cu grade for a head grade of 4.60% Cu.
- Sample 3 flash flotation and split rougher flotation open circuit recovery was 93-94% at 50-57% Cu grade for a head grade of 4.69% Cu.

The test work demonstrated that the optimized flowsheet can produce recovery of 92-95% at ±50% Cu grade for the Kakula samples tested with a head grade >4%Cu.

Figure 13.1 Optimized Kakula Open Circuit Flowsheet with Flash Flotation



Source: KCSA, 2023.

13.2 Metallurgical test work on Kamoa Resource

A number of test work 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 test work programs were completed on drill core samples of known Kamoa copper mineralization. These investigations focused on metallurgical characterization, and flowsheet development, for the processing of hypogene and supergene copper mineralization. Collectively, this body of work culminated in the derivation of a MF2 style concentrator flowsheet and performance predictions (cost and concentrate production), as applied to support the PEA (2012).

During this developmental period, the known area hosting mineralization expanded progressively, and this led to major changes to mine schedules, and associated processing schedules. As an example, over time, the supergene mineralization became less dominant, and the testing focus shifted to hypogene mineralization. Another example is that the resource and reserve grades increased, as better mineralized zones were identified. Such learning and transitions are not uncommon for this style of mineralization. The historic sample selection and test work, 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 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 summarized 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 P80 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 P80 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 recleaner 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 test work record follows.

13.2.1 Kamoa test work phase definitions

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

Table 13.1 Kamoa historical metallurgical test work

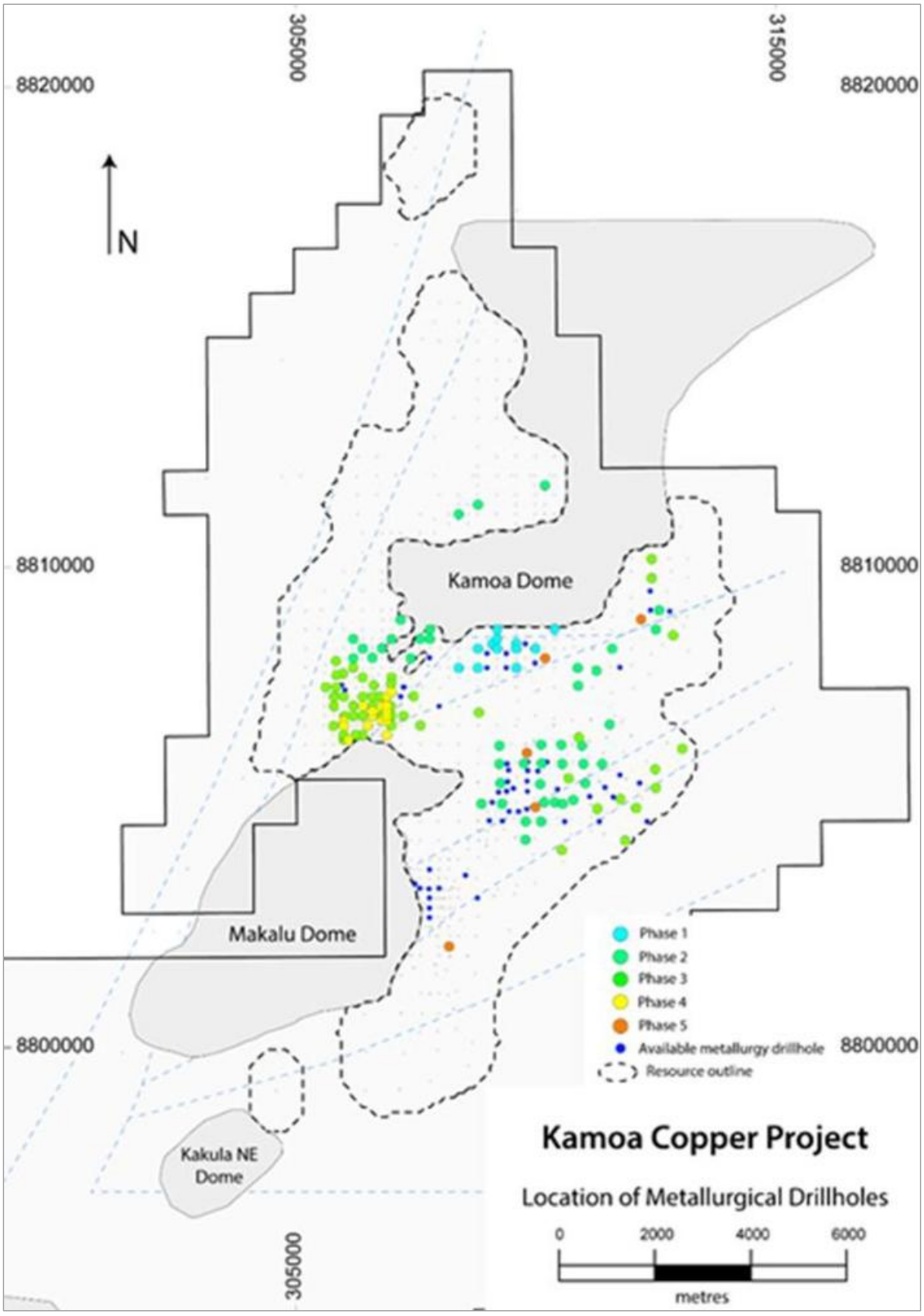
Phase	Study	Comminution	Flotation	Mineralogy	Period	Comment
1	Concept	Mintek	Mintek	SGS Johannesburg	2010-2011	Grab Samples
2	SS / PEA	Mintek	Mintek / XPS	XPS	2011-2012	Representative Composites
3	SS	Mintek	XPS	XPS	2012-2013	Composites
4	SS	Mintek	XPS	XPS	2013	Open Pit
5	SS / PFS	Mintek	XPS	XPS	2013-2014	Preliminary Variability
6	PFS	Mintek / XPS	XPS	XPS	2014-2015	Variability

13.2.2 Kamoa metallurgical sample locations

The drillhole locations that provided the historical Kamoa Phase 1-5 metallurgical samples are shown in Figure 13.2. The Phase 6 metallurgical sample locations are shown in Figure 13.3 and Figure 13.4 (red eclipses). Many of the phase samples are localized to distinct parts of the deposit, as it is now known, an indication of the evolving mine schedules.

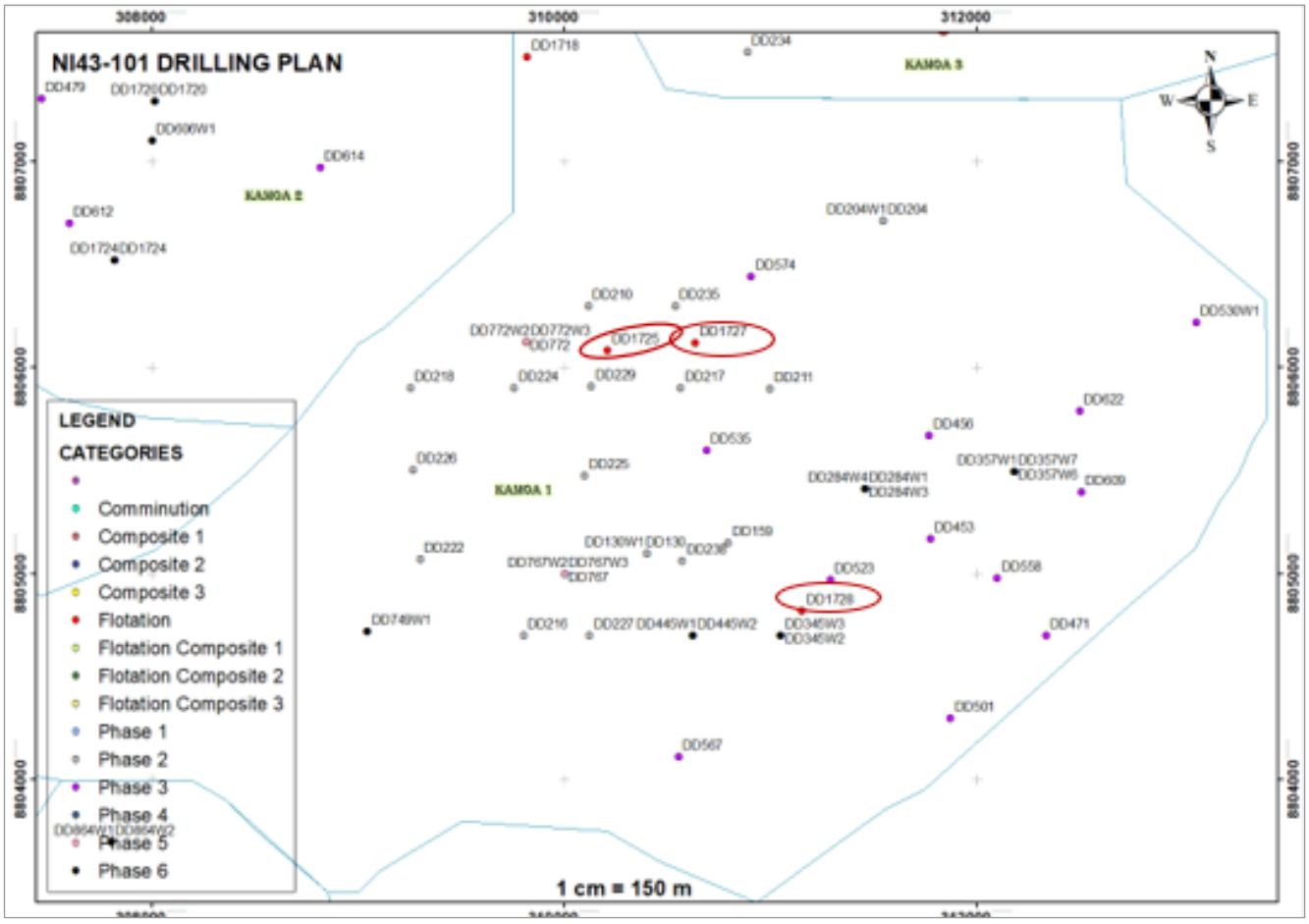
A number of the Phase 2 samples holes and a minority of the Phase 3 sample holes are in the region of the Phase 6 PFS samples. As comminution testing was carried out by area, in Phase 2, some useful information for the PFS was generated at the time. No comminution testing was conducted on Phase 3 samples,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

Figure 13.2 Kamoa metallurgical sample locations



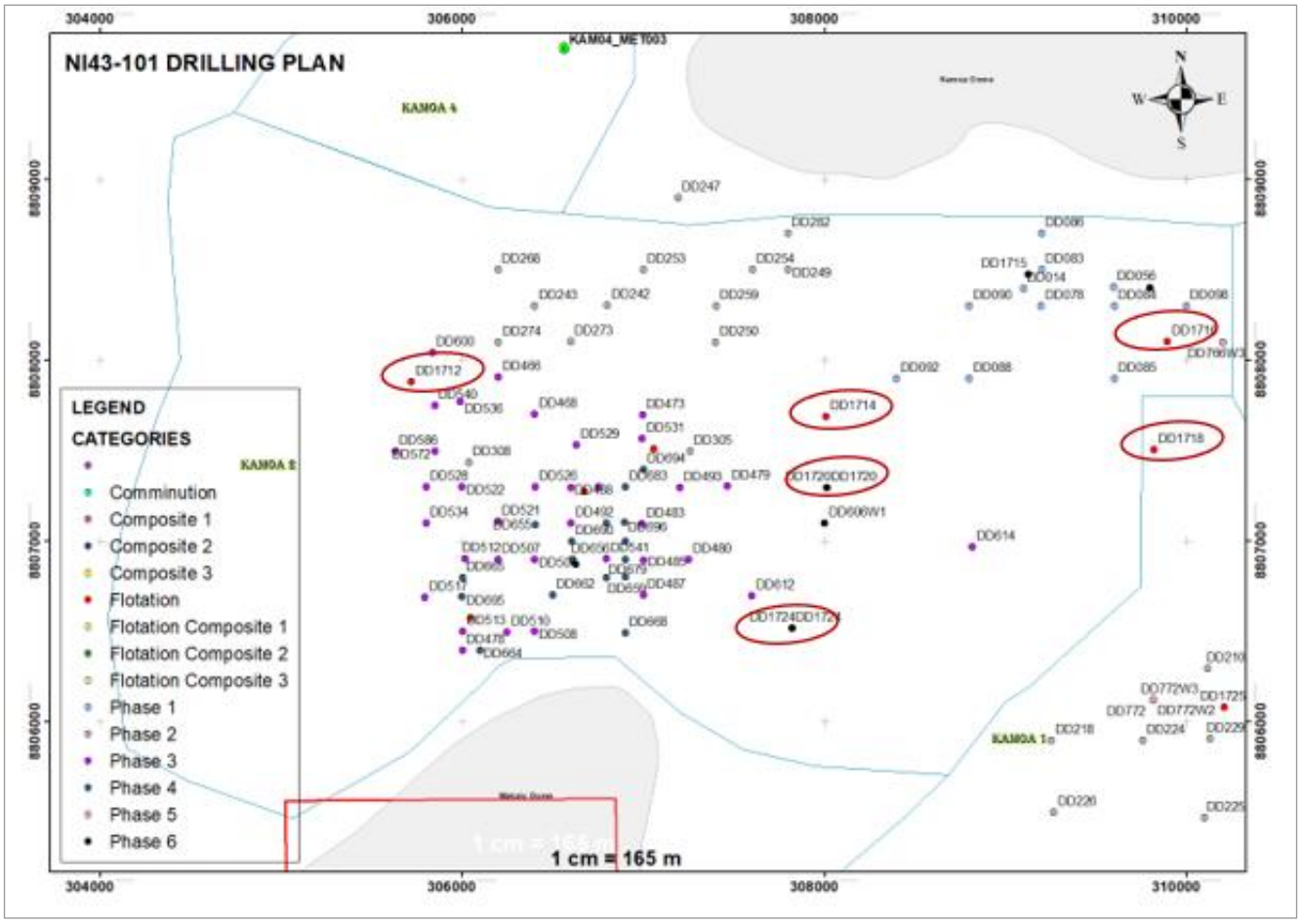
Source: KCSA 2023.

Figure 13.3 Kamoa Phase 6 metallurgical sample locations (Kamoa 1)



Source: KCSA 2023.

Figure 13.4 Kamoa Phase 6 metallurgical sample locations (Kamoa 2)



Source: KCSA 2023.

13.2.3 Kamoa comminution test work

The Phase 1-5 Kamoa comminution test programme is summarized in Table 13.2.

Table 13.2 Comminution programme, sample number 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 unmineralized)	Oxide	Pyritic Siltstone (mineralized and unmineralized, hangingwall)	Sandstone (unmineralized, footwall)
1	37–38	–	29	–
2	22–31	–	21–22	25
4	–	44–58	–	–
5	17–28	–	28	30
6				22.5

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 (hangingwall, typically pyritic siltstone). Samples from deeper in the deposit tested during Phase 2 were almost all in the extreme competence range. A reported value of Axb = 17 is amongst the most competent materials measured by the SMC method. The Phase 5 results, therefore, confirm the extreme competent nature of the Kamoa mineralization (diamictites) at depth.

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

13.2.3.2 Fine grindability summary

The BBWi test 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 P80.

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 in kWh/t range

Phase	Diamictites (Hypogene, Supergene, and unmineralized)				Oxide		Pyritic Siltstone (mineralized and unmineralized, hangingwall)			Sandstone (footwall)	
Closing Screen (µm)	212	106	75	53	106	53	106	75	53	106	53
1	–	15.5	15.7	–	–	–	16.3	14.6	–	–	–
2	–	13-17	–	–	–	–	17–20	–	–	16	–
4	–	–	–	–	11-13	11.5-14.0	–	–	–	–	–
5	20	14.5-22.0	–	13.5-21.0	–	–	15.1	–	13.4	14.5	15.2
6		16.3-20.8			11.2		15.5-17.3			18.8	

The Phase 1 and 2 samples are consistent with respect to BBWi and display slightly harder than average ball mill grindability. There is a suggestion in the Phase 2 samples that the hangingwall pyritic siltstone is harder than the diamictites. However, this is not the case with the Phase 5 samples and Phase 6 samples. The footwall sandstone sample had similar grinding properties to the diamictites. The oxidized 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 BRWi test 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 P80.

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

Phase	Diamictites (Hypogene, Supergene, and unmineralized)	Oxide	Pyritic Siltstone (mineralized and unmineralized, hangingwall)	Sandstone (unmineralized, footwall)
1	17–19	–	20.5	–
2	17–20	–	24.0	20.0
4	–	14	–	–
5	18–22	–	16.1	15.7
6	16.3-21.0	-	18.4	20.4

The diamictites for all the samples are similar, but with Phase 6 displaying some lower values than average. This is the case with 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 CWi test 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 P80 using crushing.

Note that although producing 100 µm P80 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 unmineralized)	Oxide	Pyritic Siltstone (mineralized and unmineralized, hangingwall)	Sandstone (unmineralized, footwall)
1	–	–	–	–
2	–	–	–	–
4	–	8–12	–	–
5	9–20	–	16.4	9.4
6	6-13		6-10	15-28

The crusher work indices for shallow open pit samples (Phase 4) 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 Phase 5 diamictite samples were above 18 kWh/t. The Phase 6 footwall samples have a high average CWi of 20.3 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 (g)

Phase	Diamictites (Hypogene, Supergene, and unmineralized)	Oxide	Pyritic Siltstone (mineralized and unmineralized, hangingwall)	Sandstone (unmineralized, 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
6	0.03-0.11		0.04-0.07	0.38

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 characterization summary

The four comminution properties measured are summarized in Table 13.8.

Table 13.8 Comminution summary by mineralization type

Phase	Diamictites (Hypogene, Supergene, and unmineralized)	Oxide	Pyritic Siltstone (mineralized and unmineralized, hangingwall)	Sandstone (unmineralized, 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 mineralization 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 favorable abrasiveness values in mineralized material mean the ball and liner consumptions will be low. Due care should however be taken to minimize dilution via the abrasive footwall sandstone.

13.2.4 Kamoa flotation test work

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. The tests, the first on Kamoa mineralization, were designed to confirm amenability of the copper sulphide mineralization to recovery by flotation. Samples were selected to represent what were the three important mineralized 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.

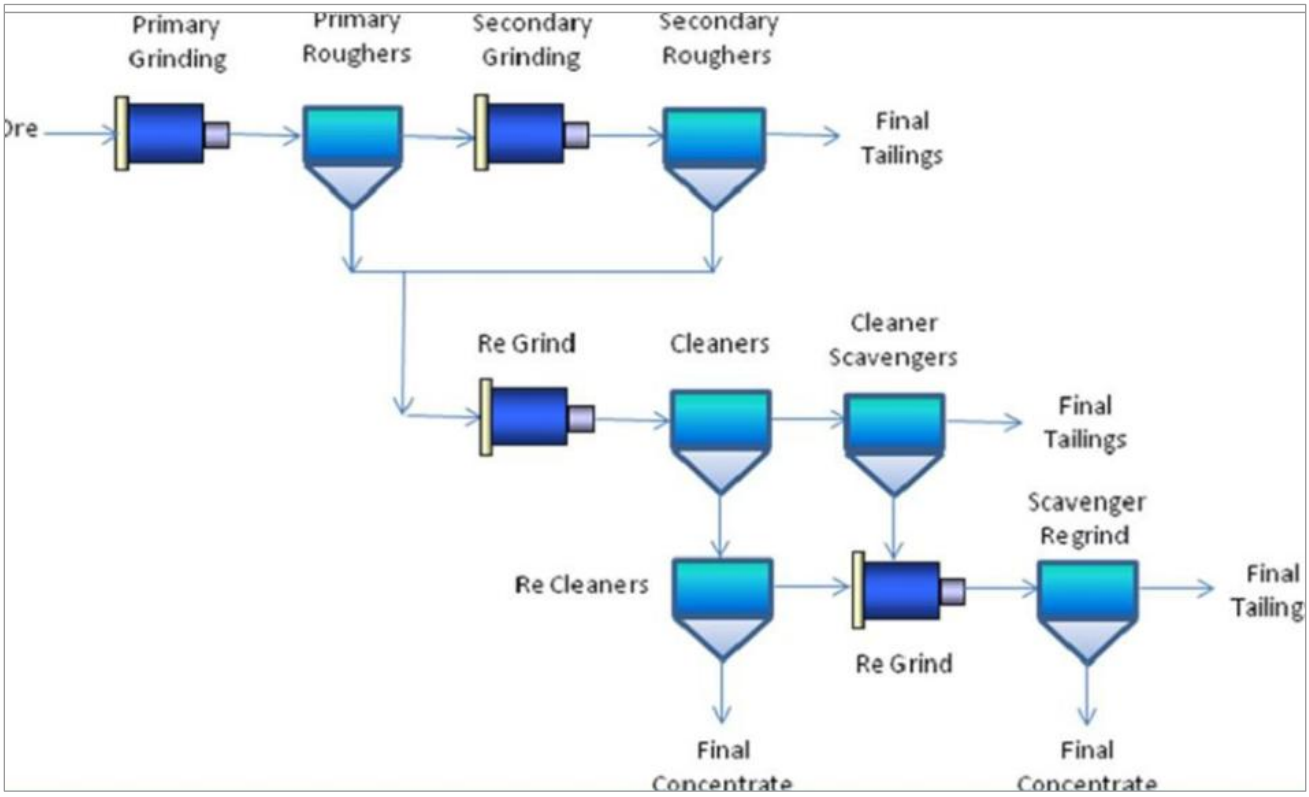
The samples were subjected to some basic bench scale testing including grinding, rougher flotation, concentrate, and tailings regrind and cleaner flotation optimization. The separation work was supported by chemical and mineralogical analyses.

This Phase 1 flotation programme indicated:

- The mineralization was amenable to treatment by conventional sulphide flotation, but with the provision that a significant amount of regrinding is required. Flotation recoveries were lower than typical Copperbelt ores due to a non-floating copper sulphide population locked in silicates at sulphide phase sizes of 10 µm or finer.

- The economic copper minerals identified include chalcopyrite, bornite, and chalcocite.
- Copper concentrate of greater than 25% Cu was achievable for both the Supergene, and Hypogene mineralization 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 13.5), which included a second stage of regrinding on middlings streams, was proposed as the go forward flow sheet concept.

Figure 13.5 MF2 dual regrind circuit flowsheet



Source: KCSA 2023.

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 mineralization 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:

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

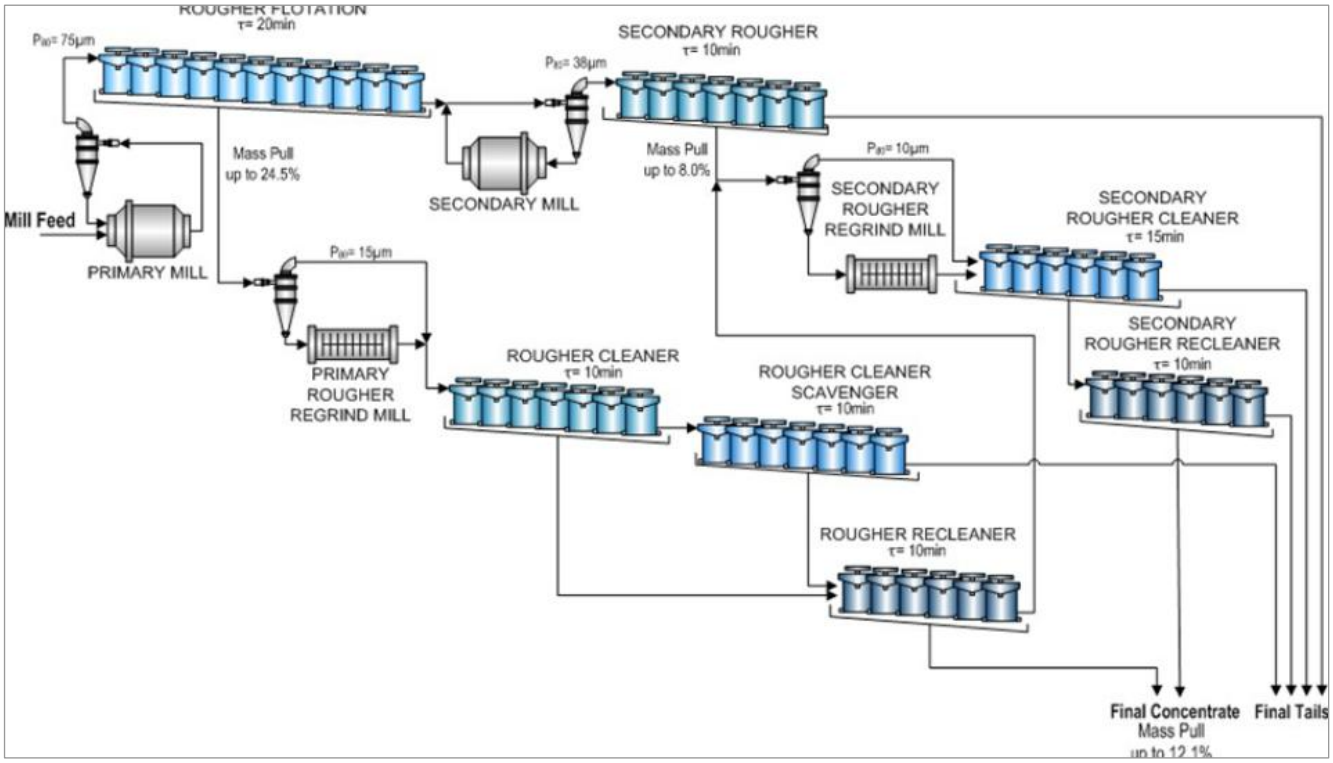
The composite samples were sized and subjected to mineralogical analysis using QEMSCAN. Parallel chemical assays were performed on the size fractions to confirm the quantitative nature of the mineralogical analysis.

Flow sheet development and optimization testing continued during this phase. A flow sheet known as the “Milestone Flow sheet” (Figure 13.6), was developed in Phase 2, that was tailored to selective recovery of the finer grained sulphide component. Similar to Mintek, the circuit relied on a mill float mill float (MF2) approach to partially liberate particles, followed by fine regrinding of concentrates to achieve a concentrate grade suitable for smelting.

Separate treatment of the primary and secondary rougher concentrates allowed for separately optimized 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 13.6 The Milestone flowsheet



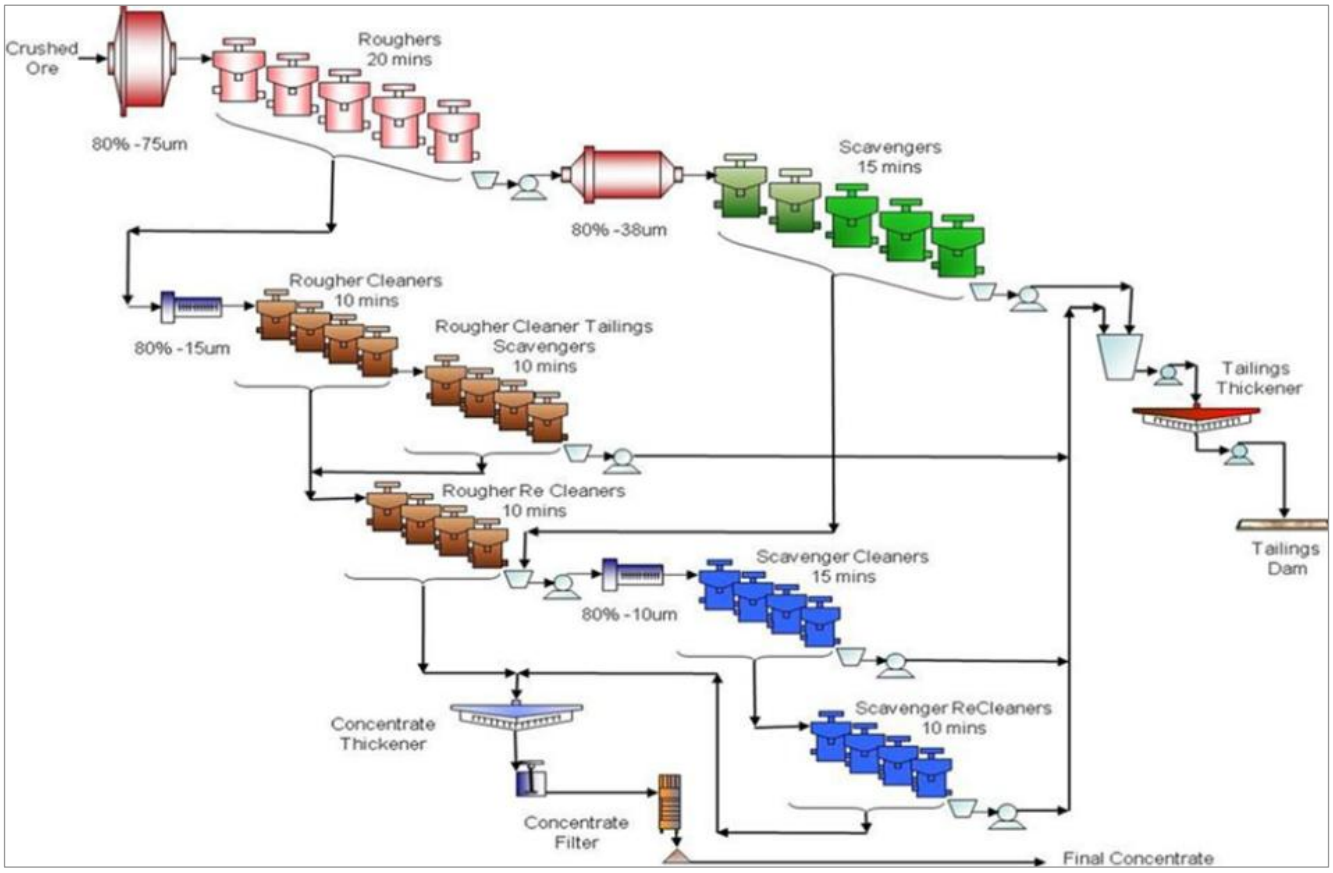
Source: KCSA 2023.

The Milestone flowsheet 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 13.7.

Figure 13.7 XPS frozen flowsheet



Source: KCSA 2023.

This Phase 3 test work programme indicated:

- Although significant differences were apparent in the copper mineralization, the samples are relatively similar in terms of gangue mineralization. The gangue minerals were dominated by orthoclase, muscovite, quartz and chlorite.
- The Supergene and Hypogene materials include a fine-grained sulphide component with more than 40% of the copper sulphide minerals having a grain size of less than 10 µm. Evidence of fine locked sulphides in silicate gangue within scavenger tails was also confirmed by QEMSCAN analysis.
- Chalcocite exhibits poorer liberation than chalcopyrite and bornite, which can lead to chalcocite losses in the scavenger tails and lower recoveries in the Supergene mineralization. However, chalcocite is often found in close association with chalcopyrite rather than gangue minerals, so that 'unliberated' chalcocite can be recovered with the other copper sulphide minerals in some cases.
- Small amounts of pyrite (3.4% and 1.3% respectively) were noted in the Hypogene and Supergene composite samples. The pyrite content was determined to have been mostly contributed from samples in the Kamoa Ouest area. This pyrite content was noted to cause acidic flotation conditions which negatively affected metallurgical performance if high chrome grinding media were not used, or if a pH modifier was not added.
- In terms of copper mineralization, 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 mineralization.

- No significant non-sulphide sulphur minerals were identified in the Supergene or Hypogene samples such that total sulphur analysis could reasonably be assumed to be equivalent to the sulphide sulphur analysis.
- Other than silica, there are no penalty elements present that reach problematic levels in the concentrate.
- Hangingwall and footwall material when mixed with the main mineralized 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 test work 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 test work 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 P80 150 µm achieved higher rougher Cu recoveries as compared to the MF1 circuit at the same grind.
- The effect of grind test work indicated that the MF1 P80 150 µm cleaner test utilizing coarser primary regrind media had a potential to achieve the target specified for the Phase 5 test work. 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 P80 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 P80 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 optimization 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 P80 of 150 µm 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 P80 was about 30% compared to the frozen flow sheet.

This excellent recovery at p80 of 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 test work

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.

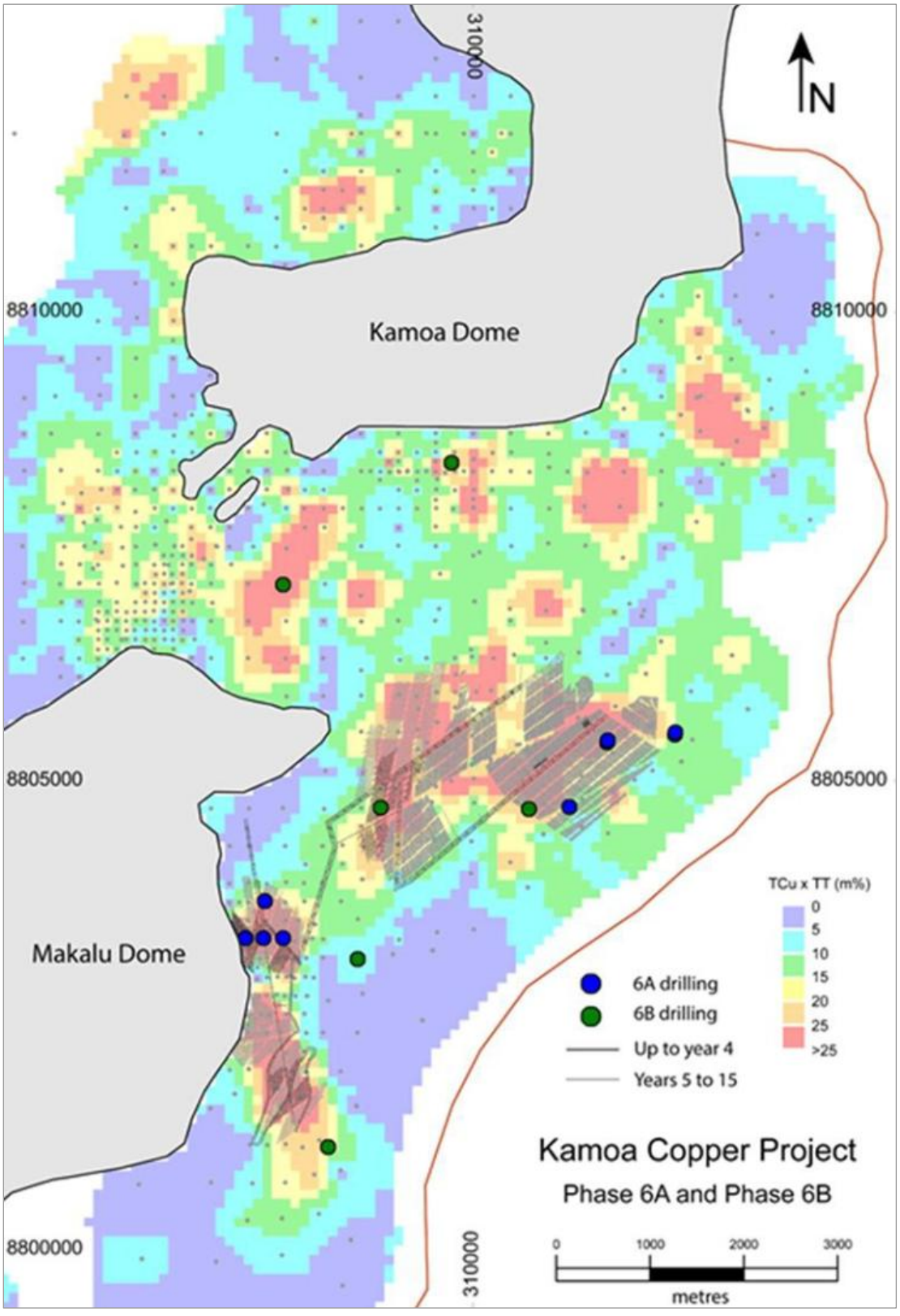
13.2.5.1 Phase 6 comminution test work - Mintek

Samples were collected for comminution testing. The samples consisted of hangingwall composites, footwall composites and variability samples from what has been termed the Minzone. Minzone refers to the single 6–12 m thick mineralized zone which is a consistent feature at all locations across the Kamoa deposit. Minzone samples have been prepared on the basis that the entire mineralized 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 Table 13.8 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 13.8 Drill collars for Phase 6A and 6B samples



Source: Ivanhoe, 2016.

Table 13.10 Comminution properties

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 hangingwall and footwall composites are each prepared from core adjacent to the seven Minzone samples and were given a weighting of seven eighths in the calculations.

Table 13.11 Comminution properties comparison

Mineralization 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 (Hangingwall)	Axb	–	21–29	-
	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.

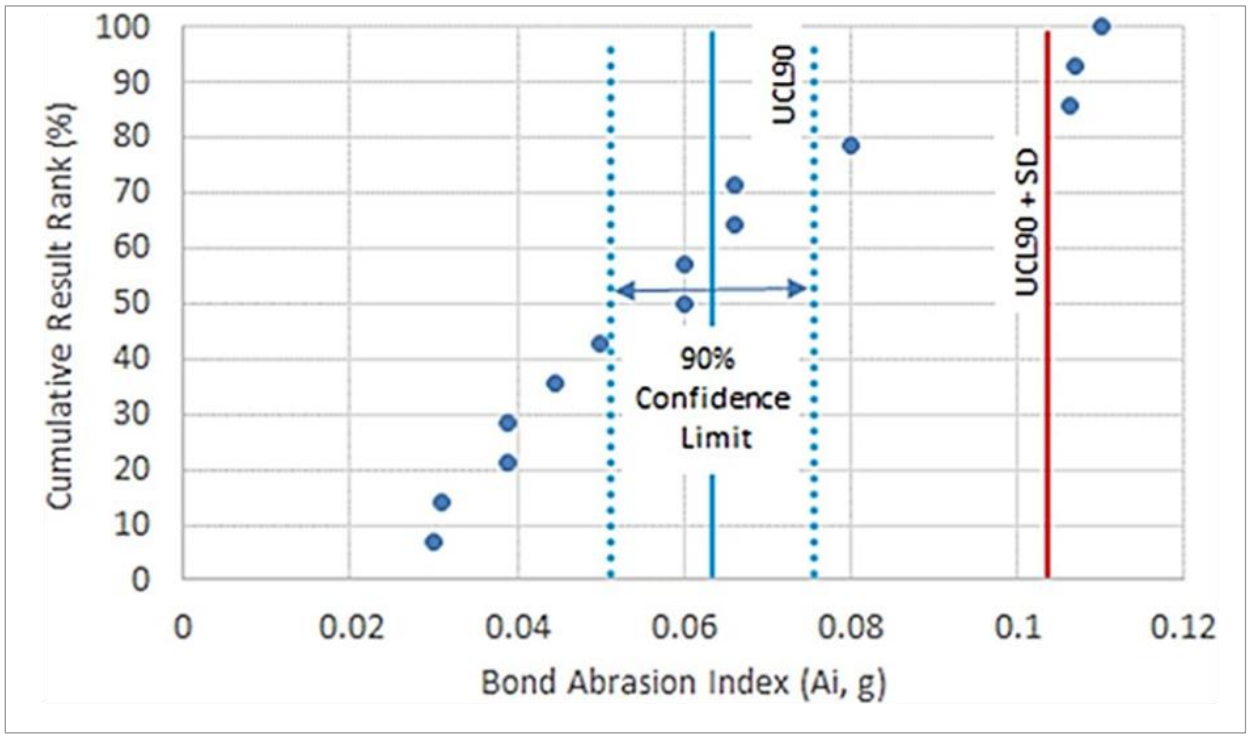
The Kamoa 2017 PFS basis of design (BOD) used the comminution properties in Table 13.12. An appropriately high CWi value has been selected.

Table 13.12 Comminution design parameters

	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 13.9. The points on the graph are the fourteen measured values for Ai on underground samples (Phases 2, 5, and 6).

Figure 13.9 UCL90 determination for Ai



Source: KCSA 2023.

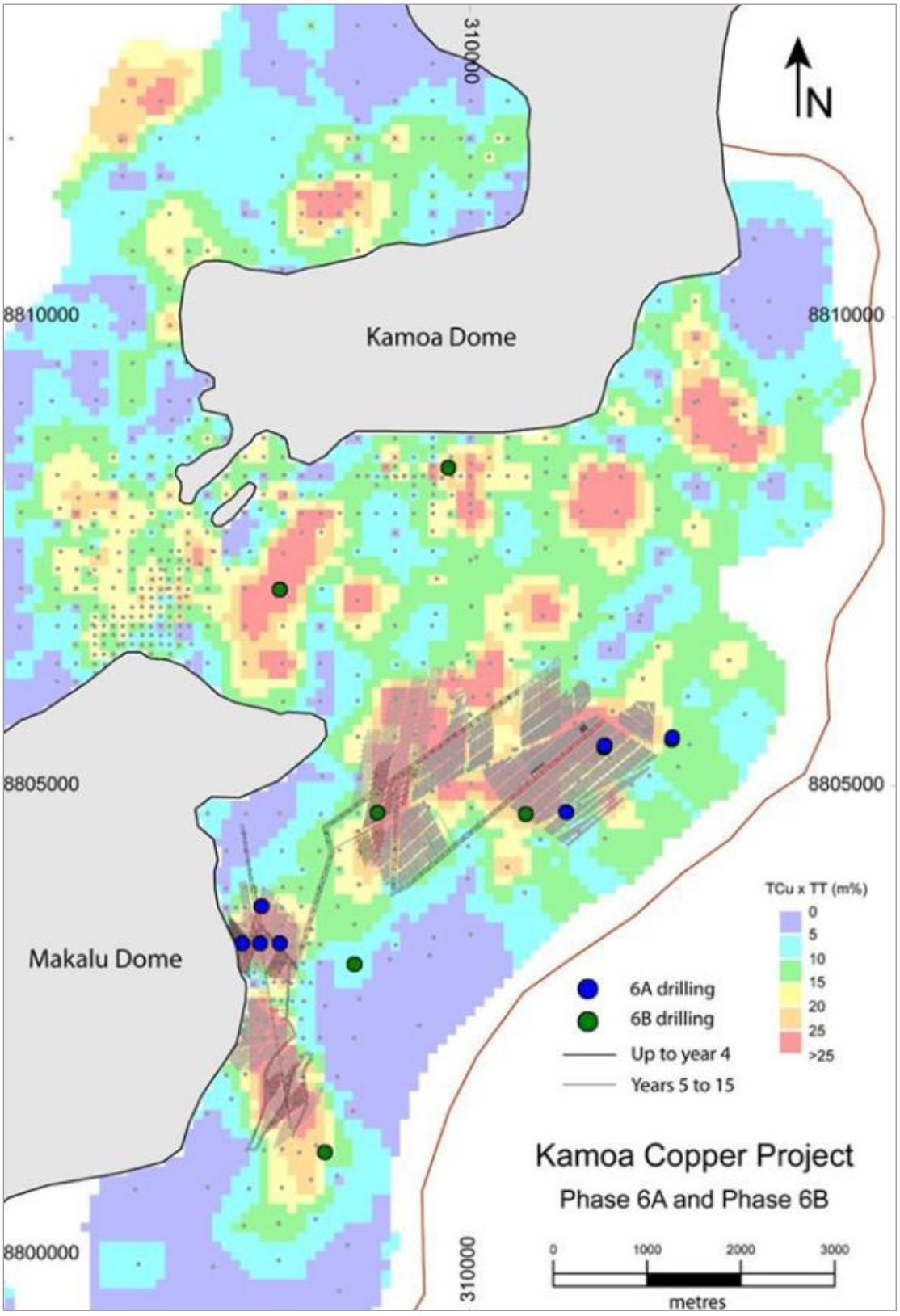
The mean value for the set is Ai = 0.063. The confidence limit is a measure of how confidently the mean or average value has been measured by the testing actually performed. As more samples are tested, the measurement of the mean value improves. Practically speaking, it means that if the same number 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 test work programme was designed to establish the performance of the preferred flotation flowsheet 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 13.10.

Figure 13.10 Drill Collars for Phase 6 flotation test composite samples



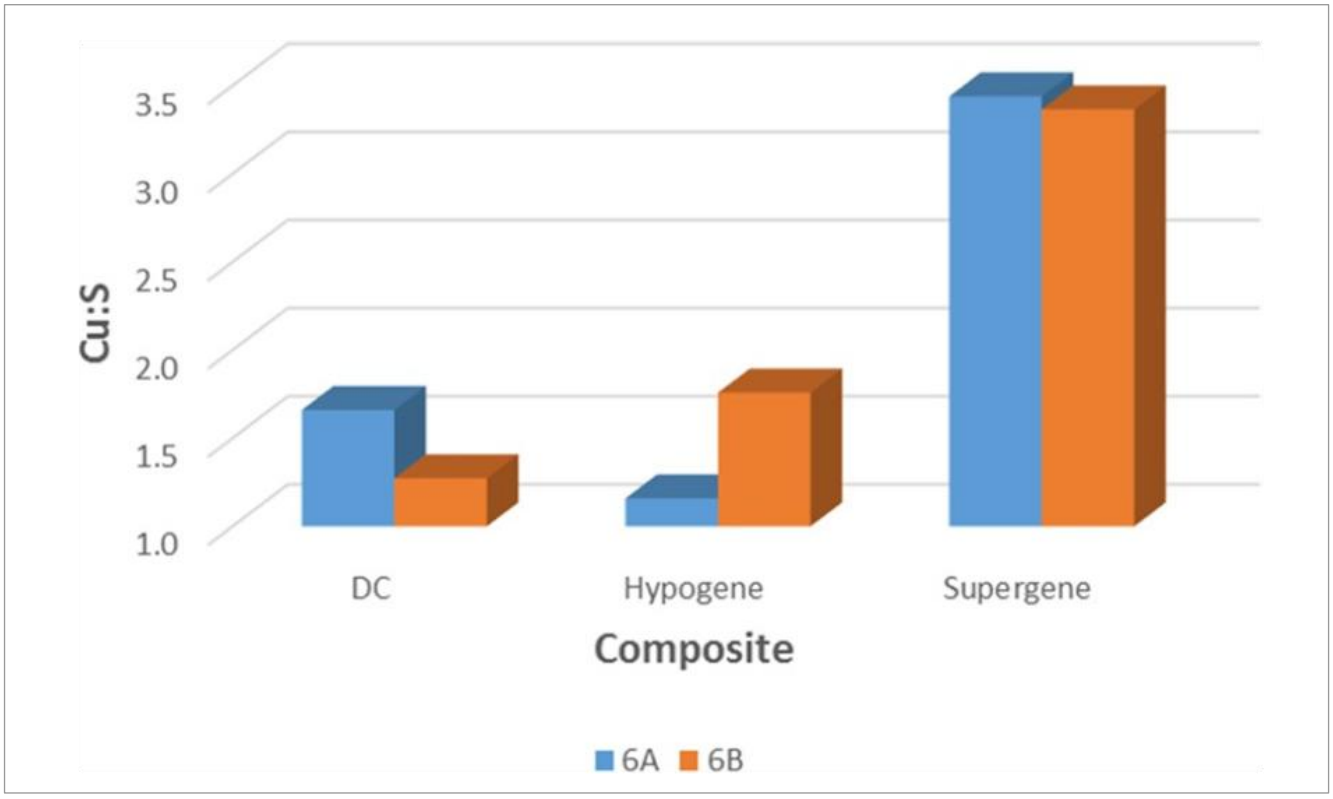
Source: Ivanhoe, 2016.

Table 13.13 Phase 6 flotation test composite feed grades

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 sulphur as shown in Figure 13.11.

Figure 13.11 Copper to sulphur ratios in Phase 6 composites



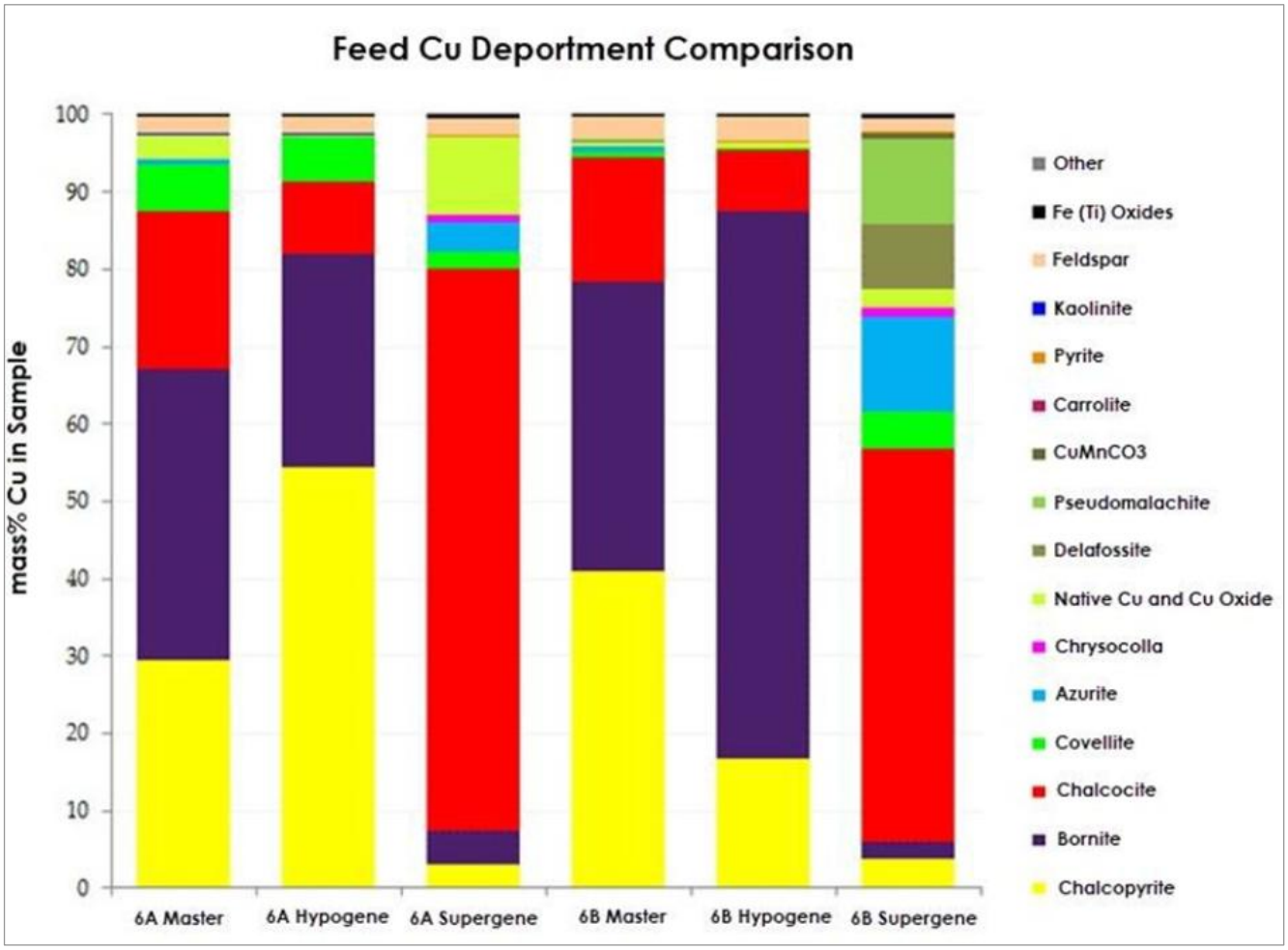
Source: Amec Foster Wheeler, 2016.

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 mineralization consists of sulphur poor copper minerals such as chalcocite and covellite as well as sulphur free minerals such as malachite and azurite. The proportions of these minerals present are clearly shown in Figure 13.2. This leads to the high Cu:S ratios shown in Figure 13.12.

The Cu:S ratio anomalies for the hypogene and supergene composites are explained by the QEMSCAN mineralogical analysis in Figure 13.12.

Figure 13.12 QEMSCAN copper mineralogy of Phase 6 Composites

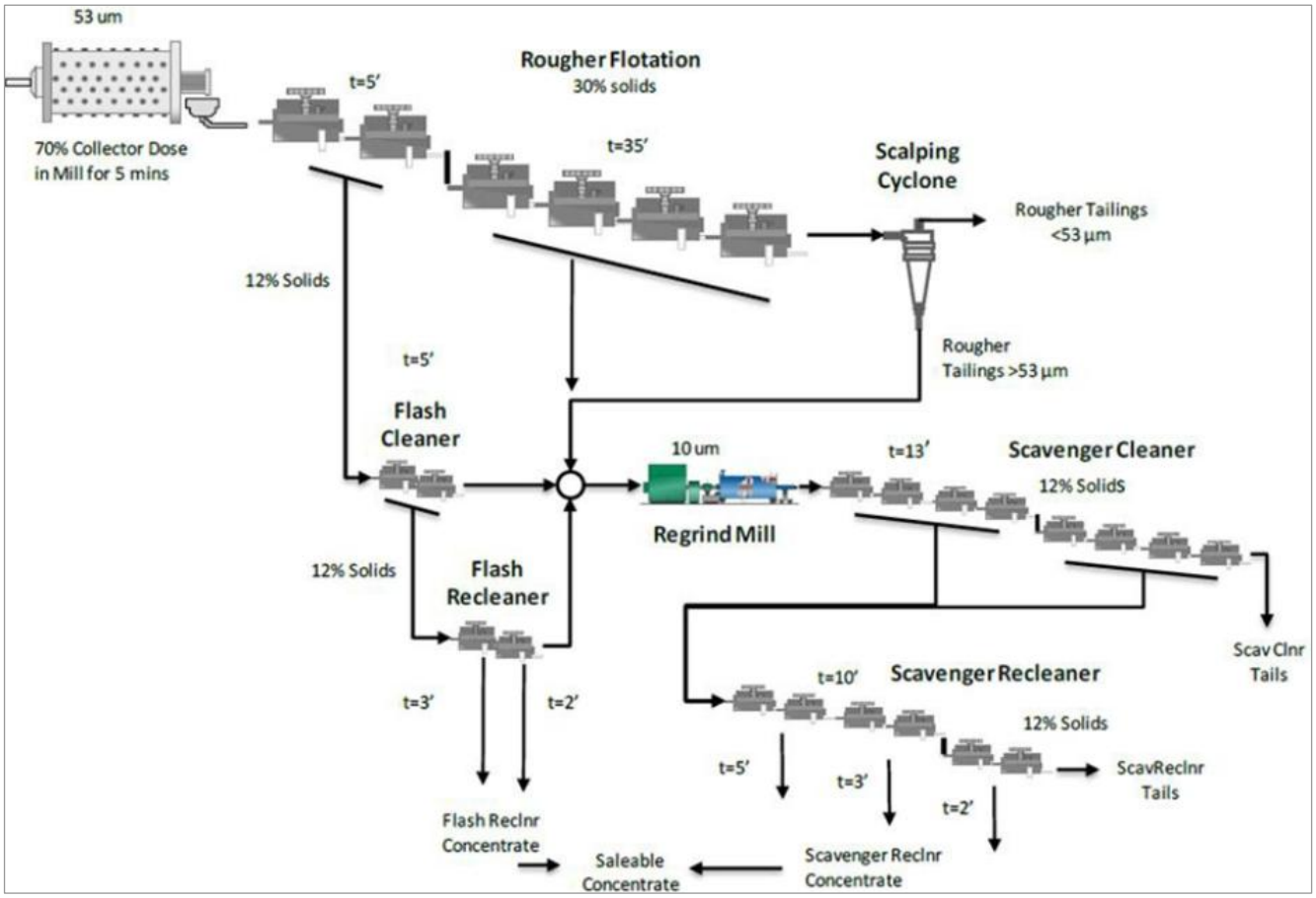


Source: XPS, 2015.

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 Flowsheet” or “IFS”. This is an MF1 or Mill Float style circuit (as opposed to the earlier MF2 circuits) and recovers both coarse (53 µm P80) and fine (10 µm P80) concentrates. The initial form of the flowsheet also has a rougher tails coarse scalping stage, a feature that did not persist into the final test flowsheet 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 13.13. Each of the six primary Phase 6 composites was tested using this flowsheet and the results are compared in Figure 13.14.

Figure 13.13 XPS IFS4 flowsheet



Source: XPS, 2015.

Table 13.14 Flotation results – IFS4 circuit

Composite		Final Concentrate					Tail	Feed
		Mass (%)	Cu (%)	Rec Cu (%)	SiO2 (%)	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 summarized in Table 13.15.

Table 13.15 Repeat 6A supergene testing – no pH adjustment to rougher flotation

Composite		Final Concentrate					Tail	Feed
		Mass (%)	Cu (%)	Rec Cu (%)	SiO2 (%)	Fe (%)	Cu (%)	Cu (%)
6A	Super	5.49	51.9	76.1	13.6	9.09	0.95	3.74

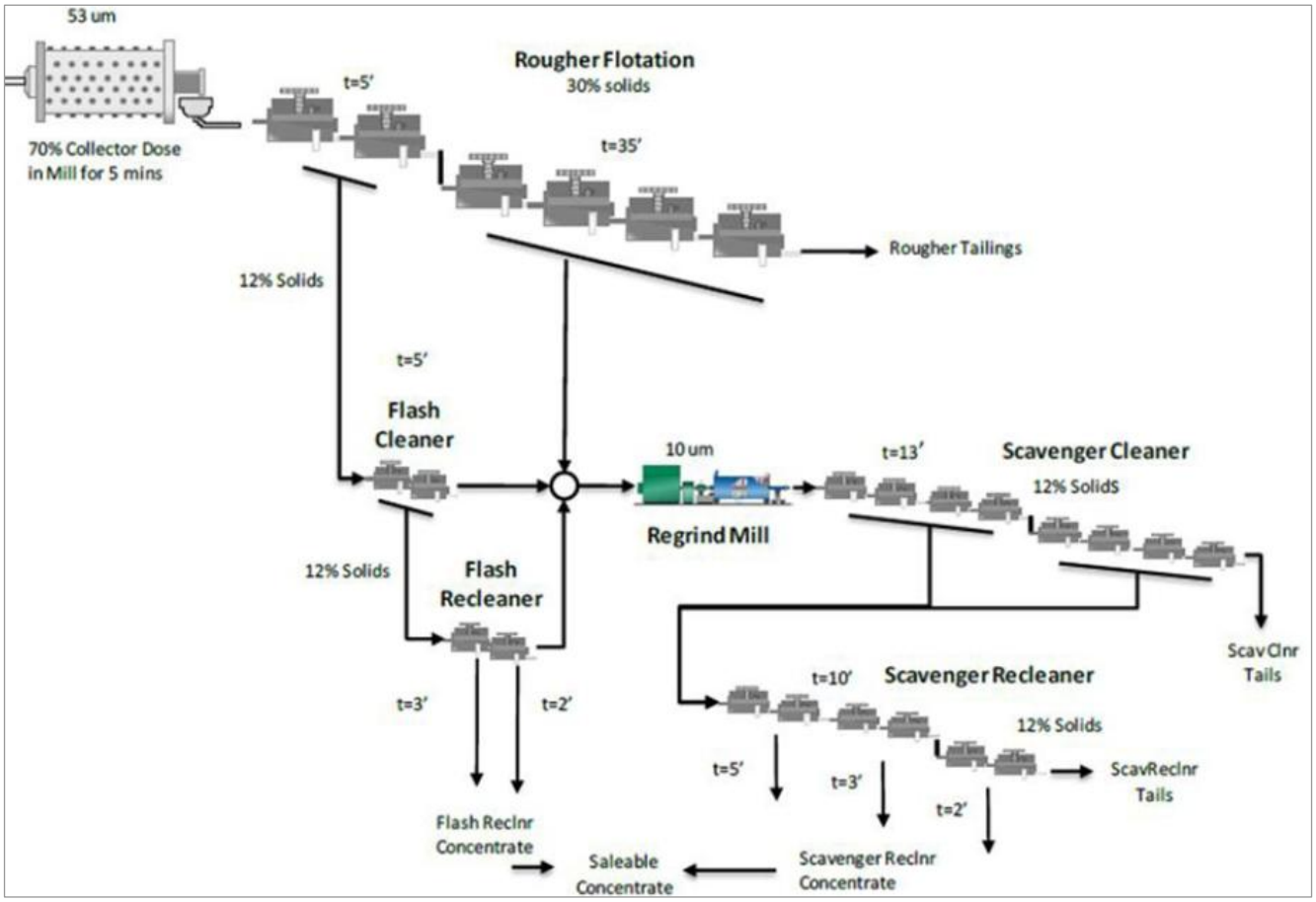
The lack of lime in the test has improved both grade and recovery for the 6A supergene sample. It is notable that the tailings grades are identical and, in general, these two results using the one sample show that the repeatability of the test is excellent.

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 test work was repeated to reflect the recommended PFS circuit. The IFS4a circuit is shown in Table 13.14.

Figure 13.14 XPS IFS4a flowsheet – basis of the Kamoa 2017 PFS



Source: 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 (%)	SiO2 (%)	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

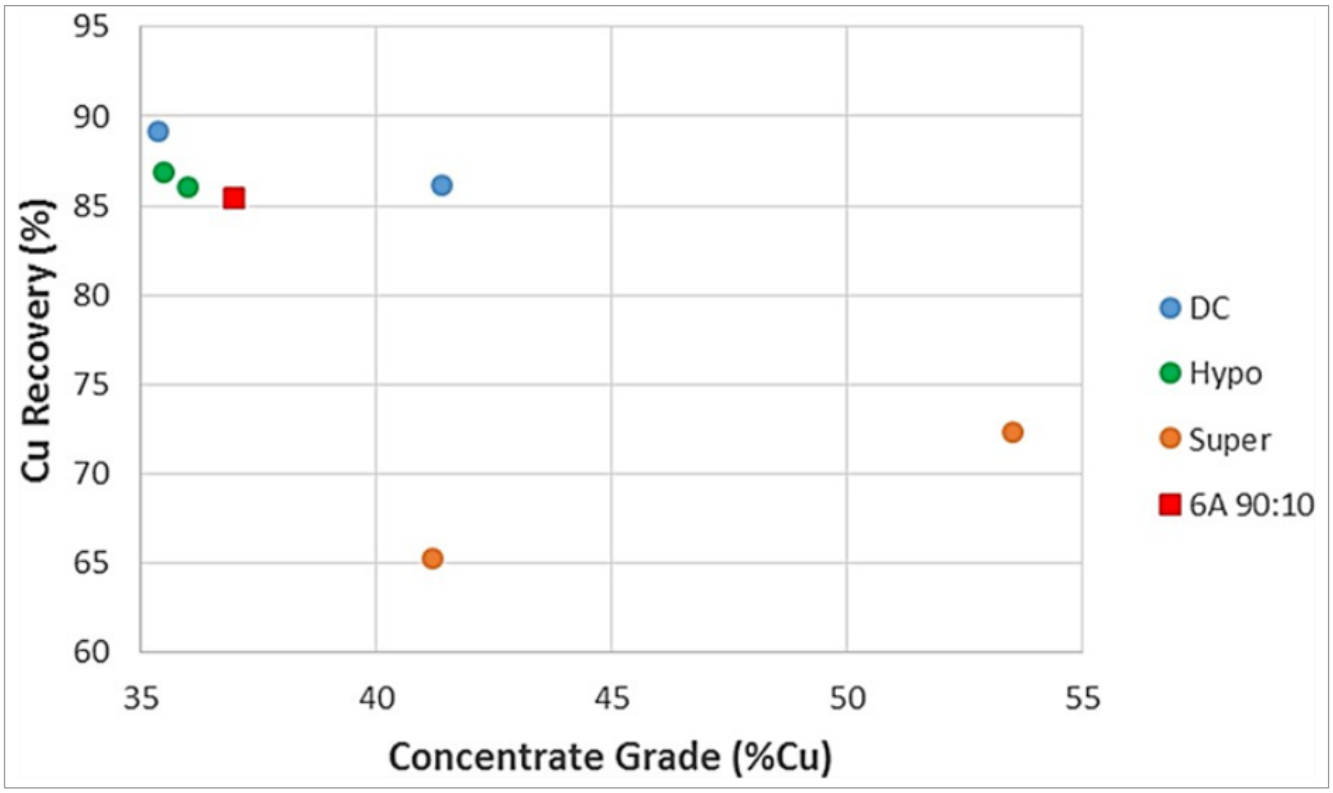
Both the IFS4 and IFS4a tests have been included in this Report to demonstrate the consistency of the test methods being used and to show the sensitivity of copper recovery to the amount of fine grinding employed.

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 Table16 has been plotted in Figure 13.15.

Figure 13.15 Recovery vs grade plot for Phase 6 IFS4a comparative flotation tests



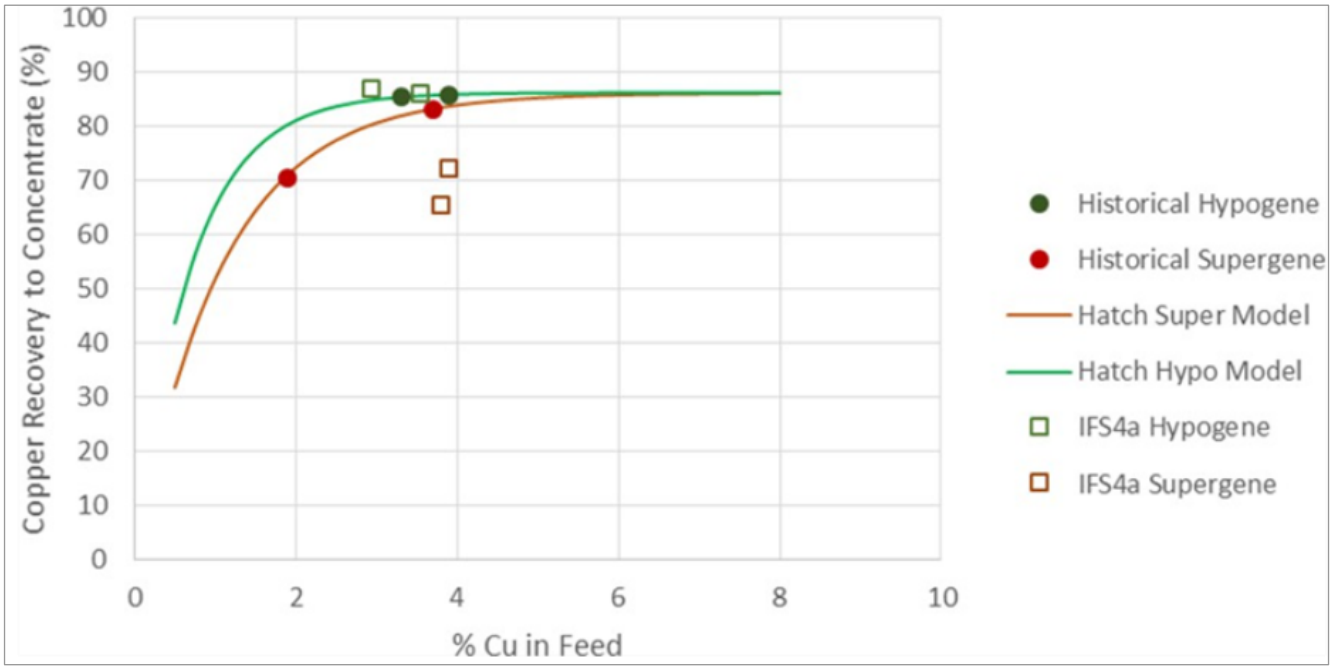
Source: Amec Foster Wheeler, 2016.

As expected, hypogene samples generate relatively low concentrate grades with good recoveries. The supergene samples generate much higher-grade concentrates but at a significant recovery penalty. The recovery loss is due to copper being present in non-sulphide copper minerals.

13.2.5.3 Copper recovery vs head grade model

To allow the prediction of copper recovery in the block model (mine planning) it is usually necessary to develop a model relating copper recovery to head grade. The recovery model from the previous Technical Report is presented in Figure 13.16, together with the performance seen in the Phase 6 IFS4a tests.

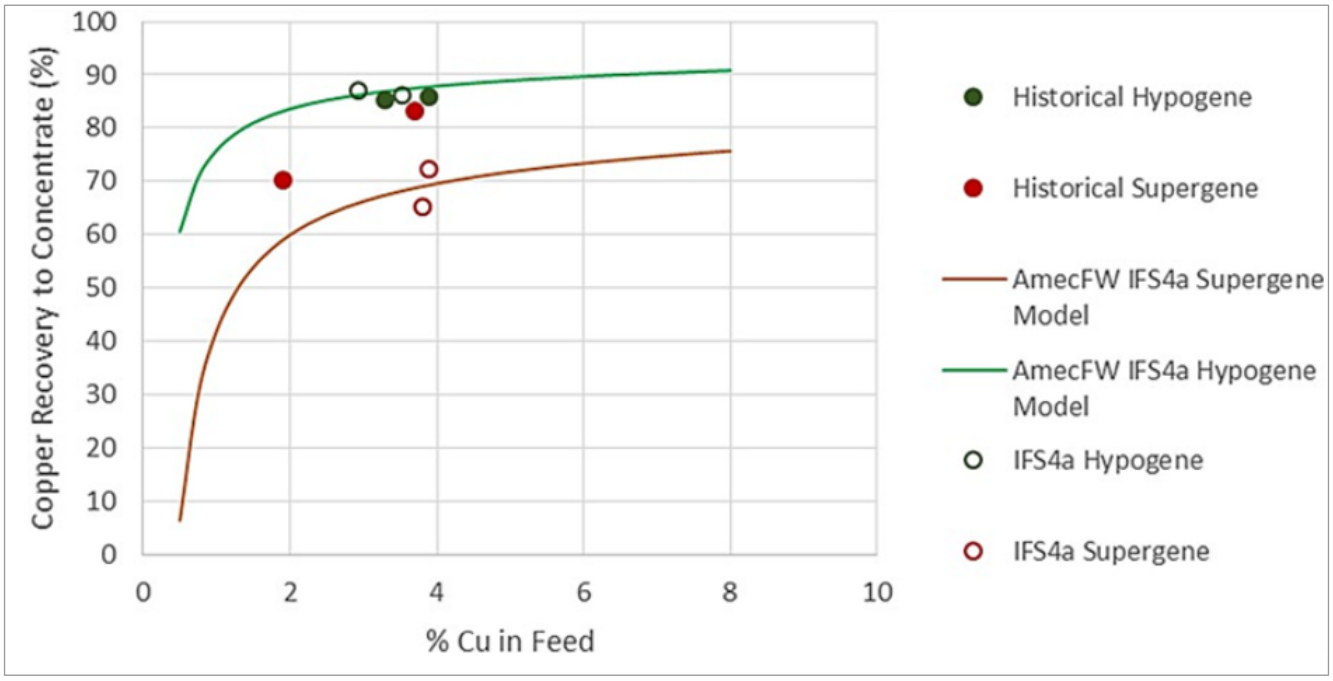
Figure 13.16 Old copper recovery model (TR 2013)



Source: 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 13.17, the new model better represents the Phase 6 results. The new hypogene results were also modelled with less recovery drop of below 3% Cu.

Figure 13.17 Updated recovery models based on 2017 PFS testing



Source: 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

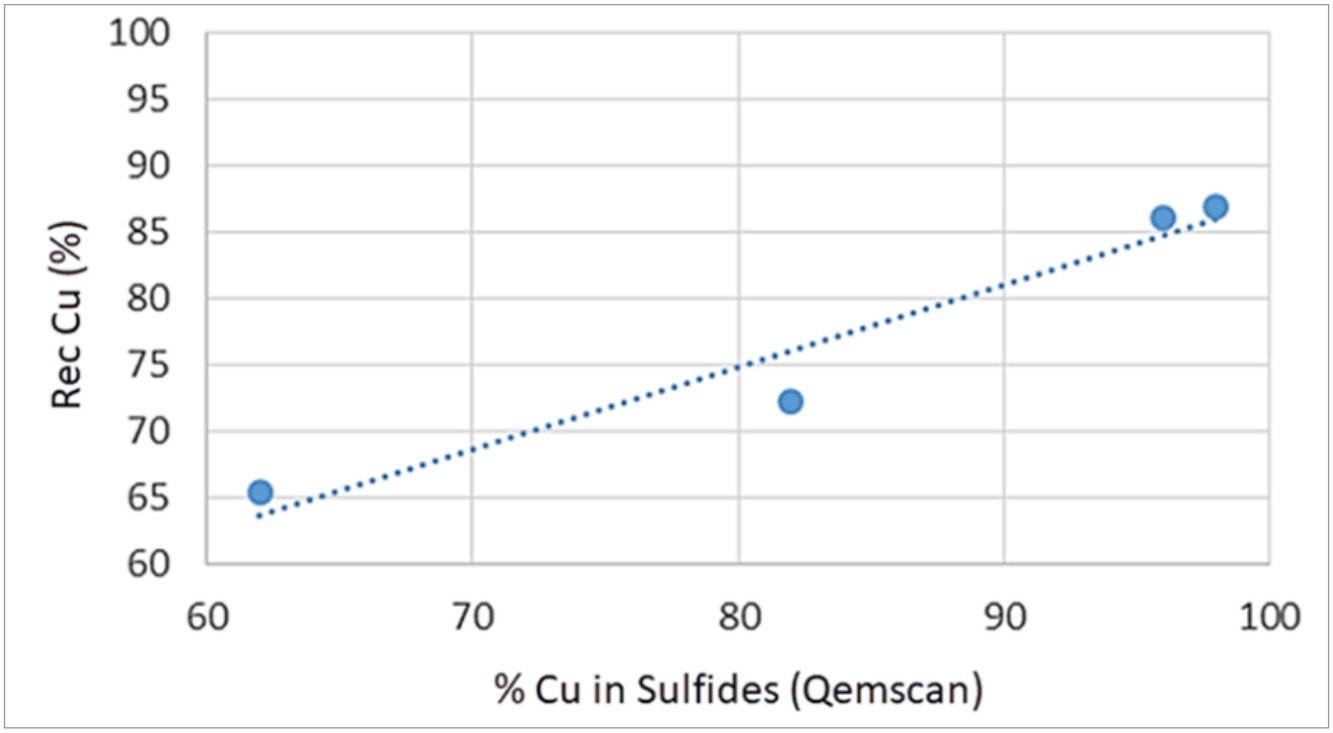
It is clear from Figure 13.17 above that supergene recovery is not well defined when it is necessary to rely on a single dependency, in this case the copper head grade. There will be a recovery relationship with head grade, but the analysis shows that the recovery is more dependent upon the proportion 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 mineralization type. It is only necessary, at this stage of the project, to modify recovery in mineralized 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 mineralized 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 sulphide mineralization, especially chalcocite, is dissolving during the ASCu determination, but this is yet to be confirmed).

For the Phase 6 test work 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 13.18.

Figure 13.18 Variation of recovery with floatable copper



Source: Amec Foster Wheeler, 2016.

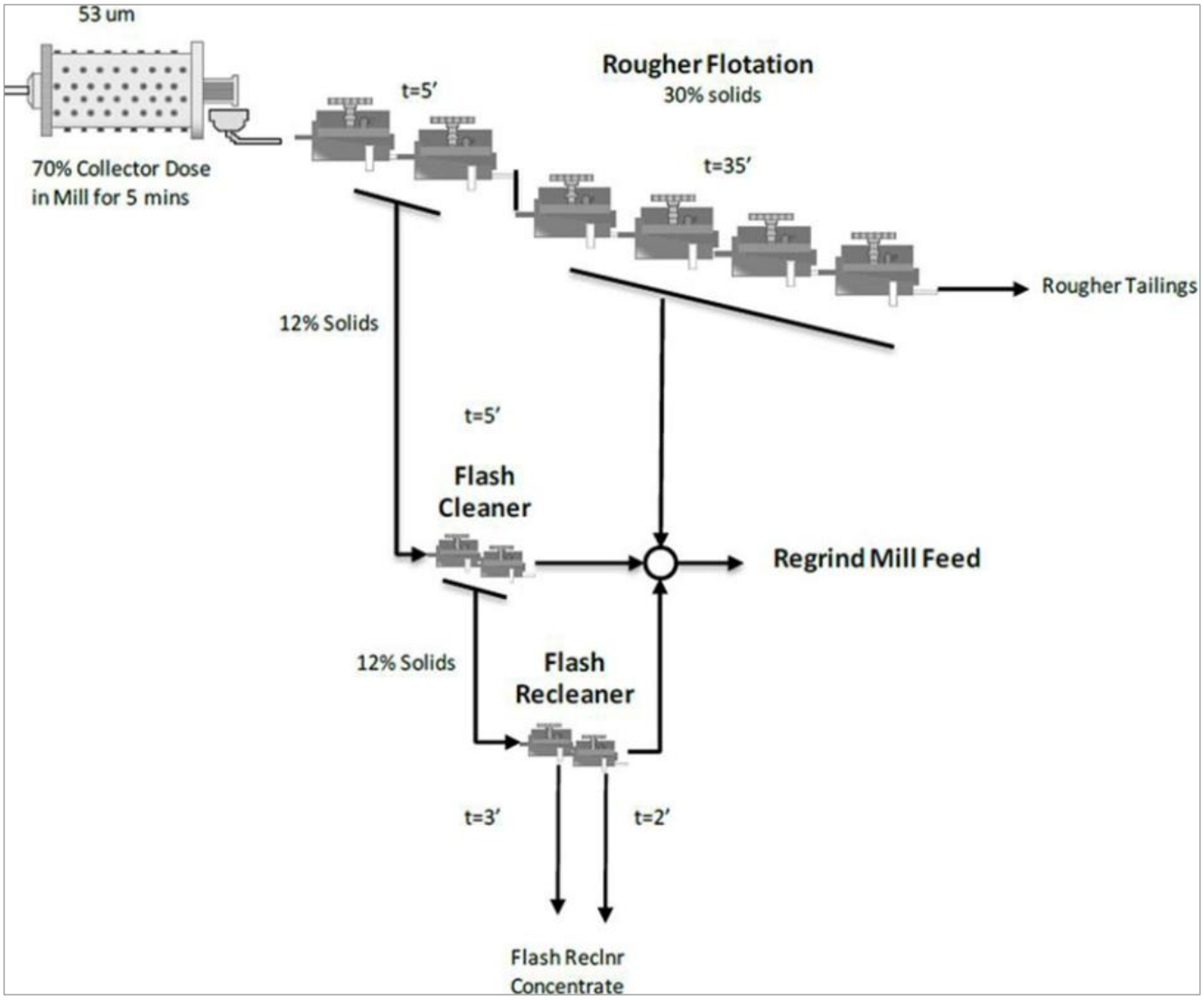
This strong relationship between recoverable copper and copper in sulphides is expected. Almost all oxide copper minerals, together with native copper, are not readily floated in a standard copper sulphide flotation chemical environment, which uses relatively low concentrations of selective collectors.

The recovery models have been updated using historical data and benchmarking with operational data. These focus on relationship between recovery and head grade.

13.2.5.4 Phase 6 test work – 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 13.19.

Figure 13.19 Truncated XPS IFS4a circuit



Source: XPS, 2015.

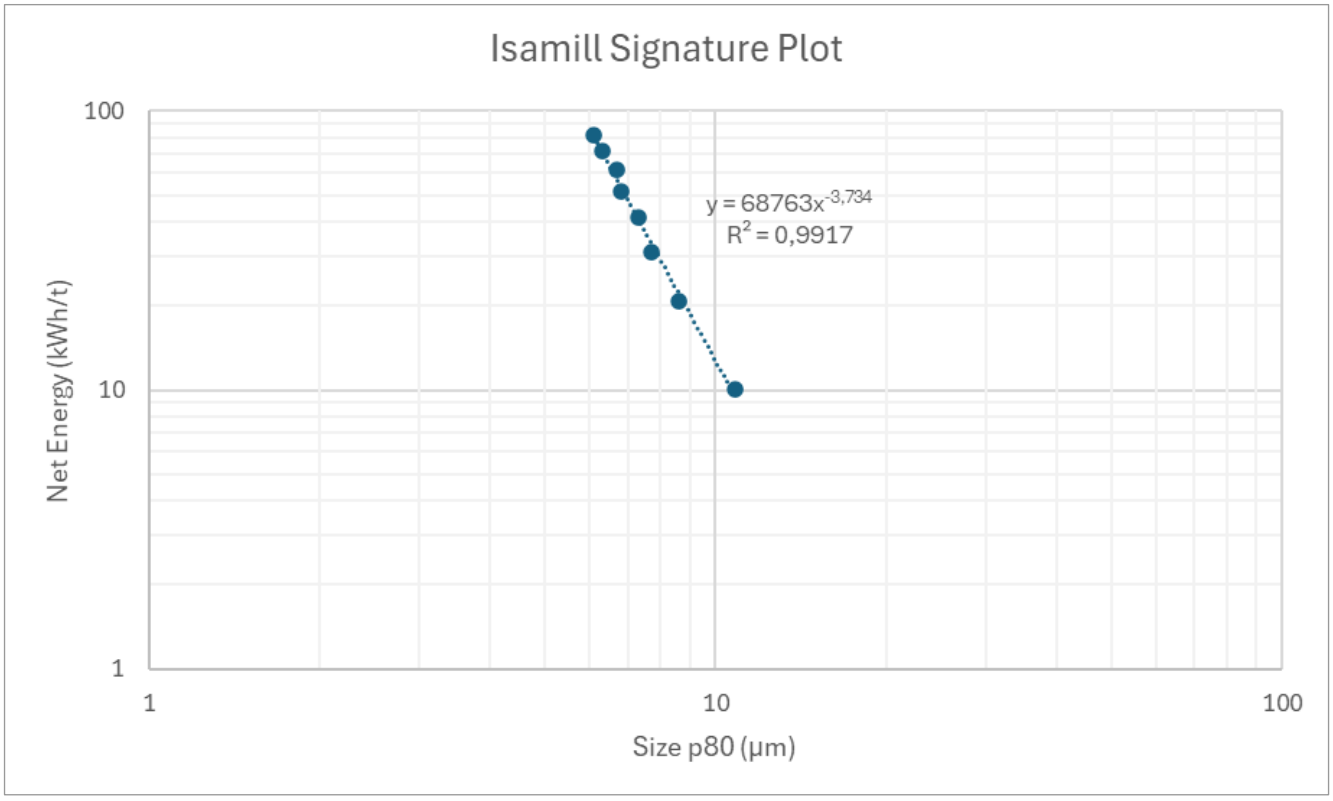
The 6A signature plot composite was prepared separately from the other composites and contained 4.35% Cu. The Cu:S is 1.37 compared to 1.66 for the 6A DC sample indicating a greater proportion of chalcopyrite in the copper mineral suite of the new composite.

Although the rougher feed was ground to a P80 of 53 µm, the regrind mill feed was much finer with a P80 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, each time increasing the energy input by 10 kWh. Samples of the product were taken at each pass. The resulting signature plot is shown in Figure 13.20.

Figure 13.20 Isamill Signature plot

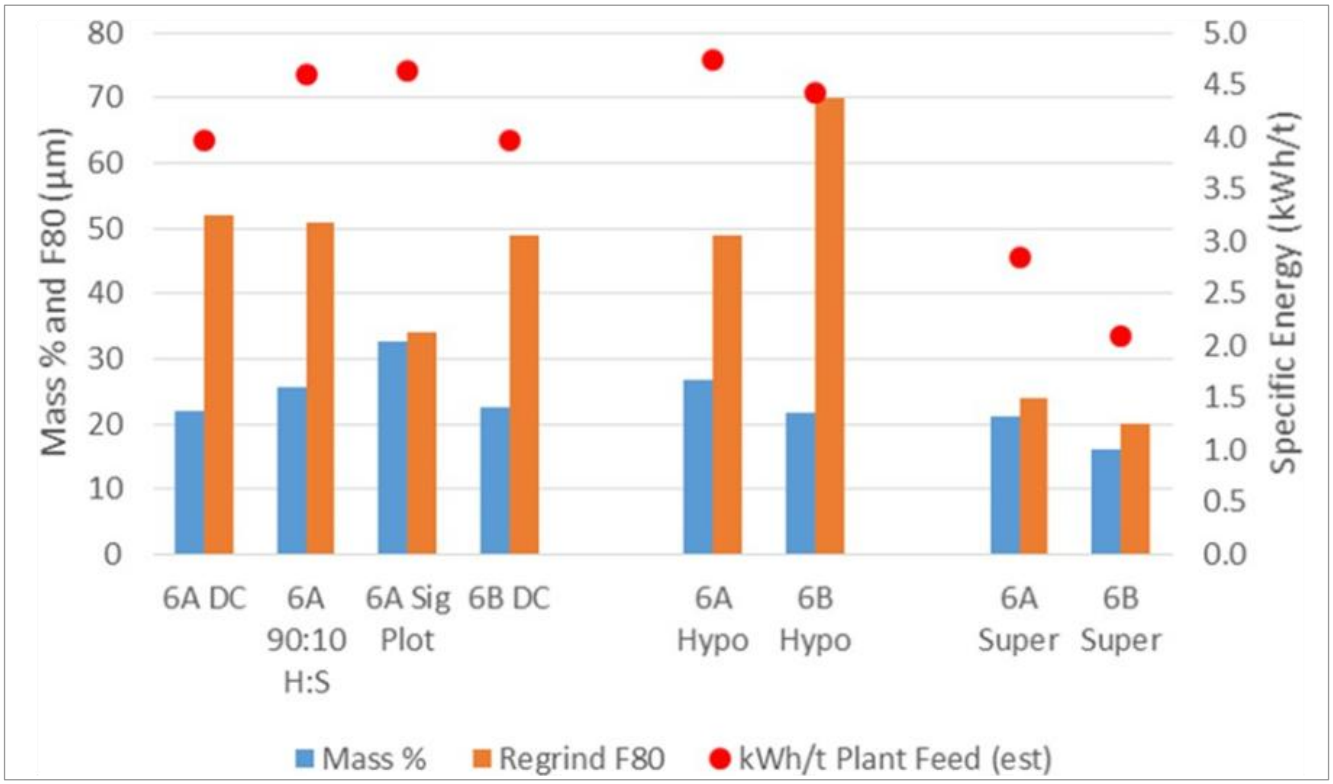


Source: XPS, 2015.

This result is based on the sample tested, and the specific grinding energy requirement for other feeds will be dependent upon the P80 of the regrind feed and the mineralogy of the feed. The sample tested required 12.7kWh/t to reduce the feed size from f80 of 35 µm to p80 of 10 µm. At a mass pull of 30.8% this equates to 3.9kWh/t treated. The specific energy required will increase for a coarser feed to the regrind mill.

An analysis of the various Phase 6 tests showed that these factors, together with the mass pull to be reground, vary considerably as summarized in Figure 13.21.

Figure 13.21 Phase 6 regrind feed variability



Source: Amec Foster Wheeler, 2016.

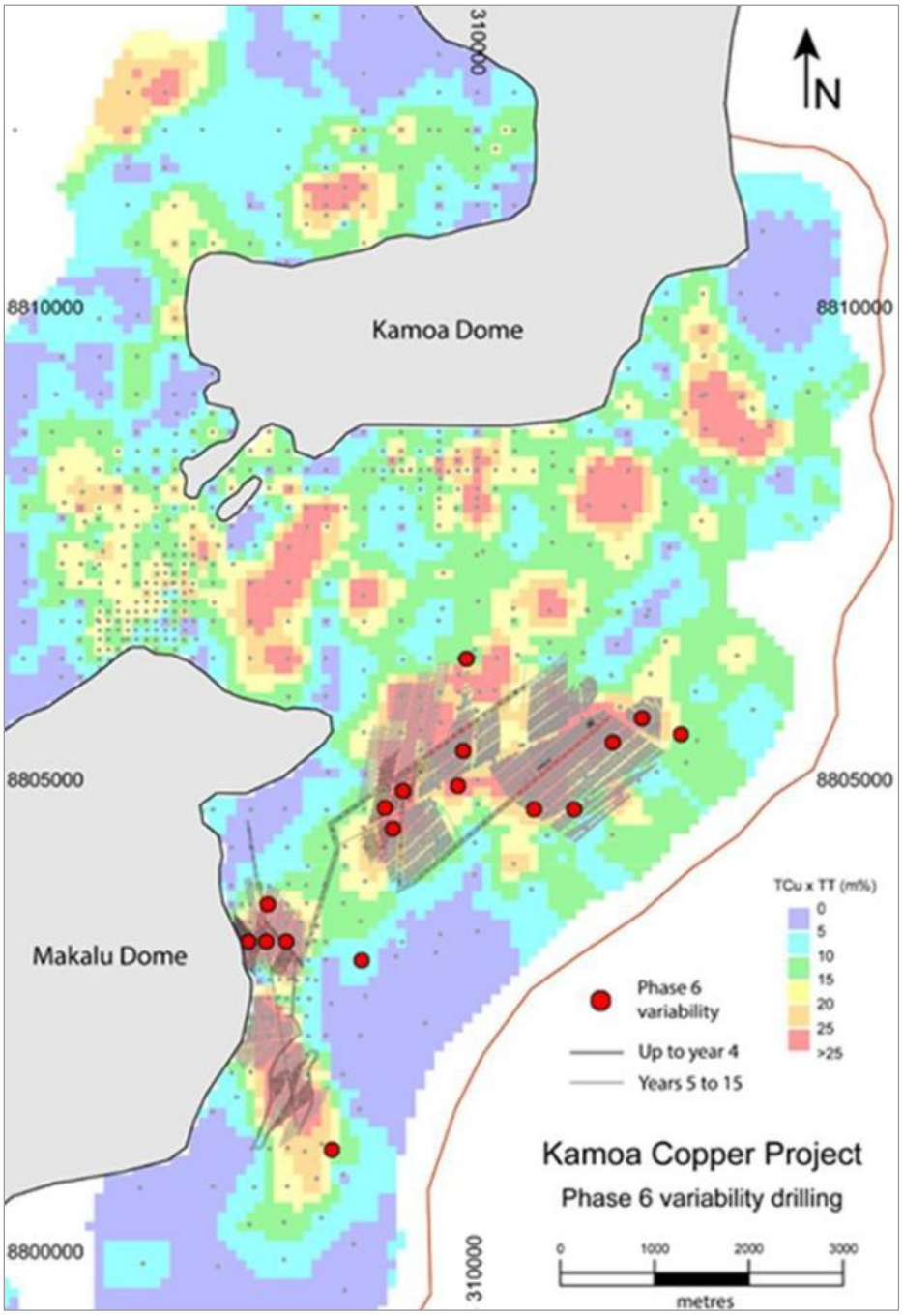
Interestingly, across the four development composites and two hypogene samples the energy per tonne of plant feed is somewhat independent of the test. This is because low mass pulls tend to have coarse particle sizes while high mass pulls are finer. From the Figure 22 data, a regrind power selection 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 test work

A programme of variability test work was completed for Kamoa using the samples indicated in Figure 13.22, together with the Year-0 to Year-15 PFS mining areas.

Figure 13.22 Phase 6 variability samples



Source: Ivanhoe, 2016.

The variability sample selections provide good spatial representation of plant feed during the proposed Kansoko mine plan period.

13.2.5.6 Kamoa copper mineralogy

The Kamoa copper sulphide mineralization exists in two basic modes regardless of copper sulphide mineral. Coarse copper sulphides, some in the centimetre size range, are clearly visible in the core. Many

intermediate sized copper mineral grains are usually visible but any that are clearly distinguishable can be considered coarse. The second mode of occurrence is a pervasive “fog” of ultrafine copper sulphides throughout the matrix.

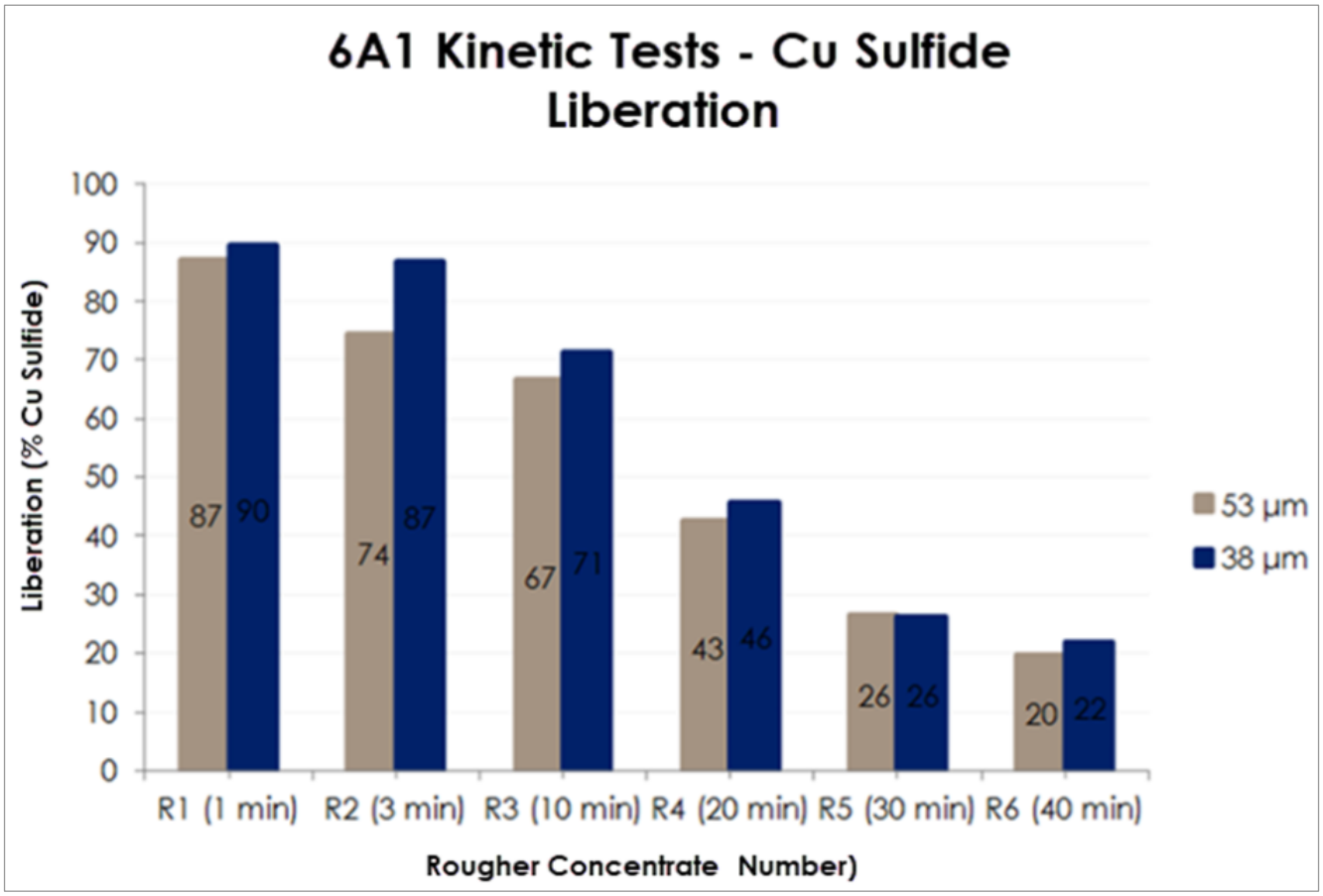
In the image below (Figure 13.23) can be seen a 2 cm wide white clast within the grey diamictite matrix, against which chalcopyrite has “mantled” during the sulphide deposition phase. In the surrounding rock matrix there are smaller mantled clasts and visible blebs of chalcopyrite (and other sulphides). What cannot be seen in the photograph is the dispersion of 1–10 µm (0.0001–0.001 cm) copper sulphides present throughout the grey matrix.

Figure 13.23 Typical Kamoa hypogene mineralization in diamictite

Source: 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 P80 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 summarized in Figure 13.24.

Figure 13.24 Copper sulphide liberation in rougher flotation

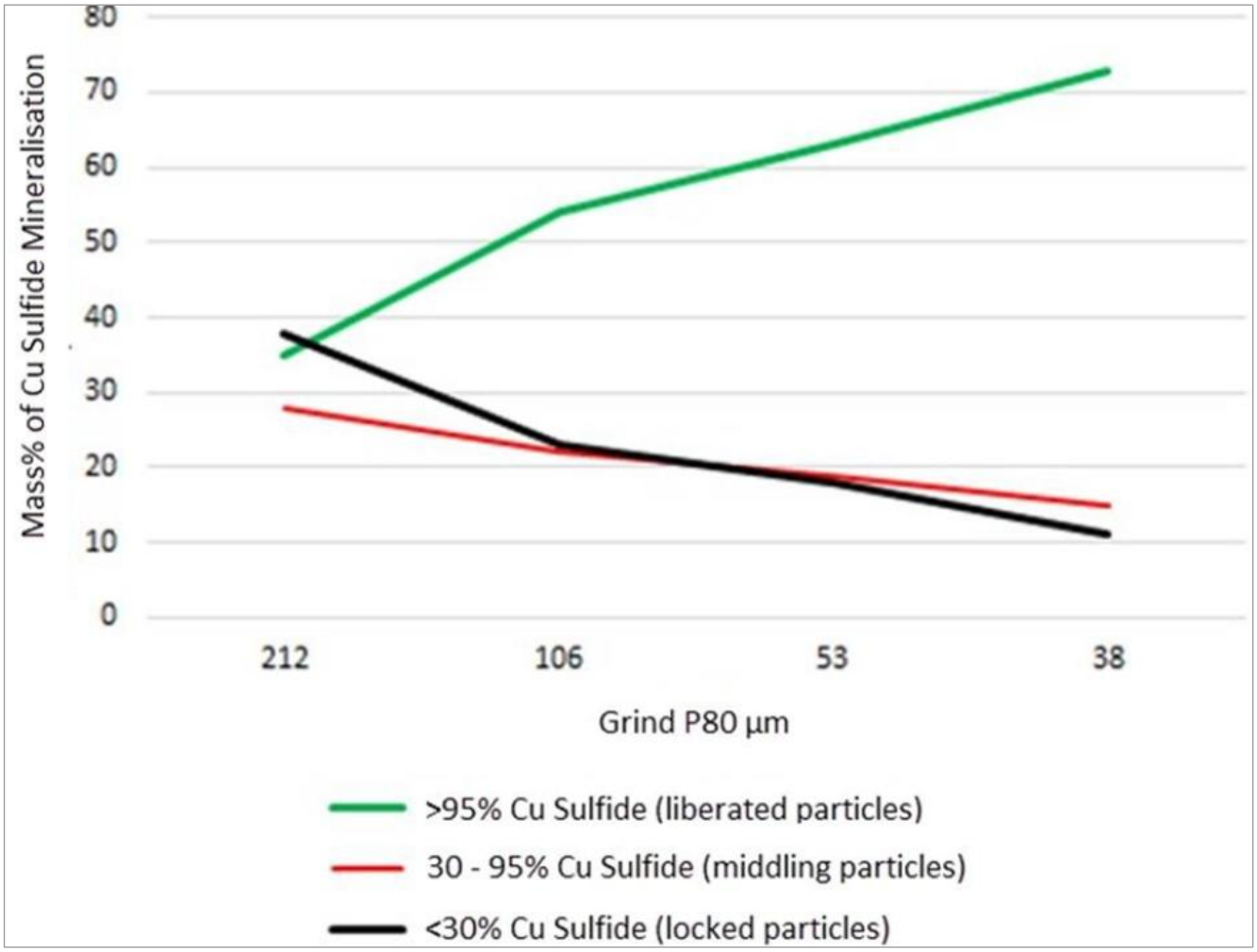


Source: XPS, 2015.

The highly liberated copper sulphides are floated preferentially while the poorly liberated sulphides float towards the end of the test. It is also clear that at the finer grind size (+38 µm) the overall liberation level is higher than in the 53 µm test.

Copper sulphide morphology in all Kamoa, and Kansoko samples, is consistent in that the minerals are always present as both very coarse, and very fine, grains. The large proportion of copper in fine sulphides is the reason for the strong liberation effect of grinding (measured using QEMSCAN, XPS Laboratory) as shown in Figure 13.25.

Figure 13.25 Phase 6 hypogene composite liberation analysis

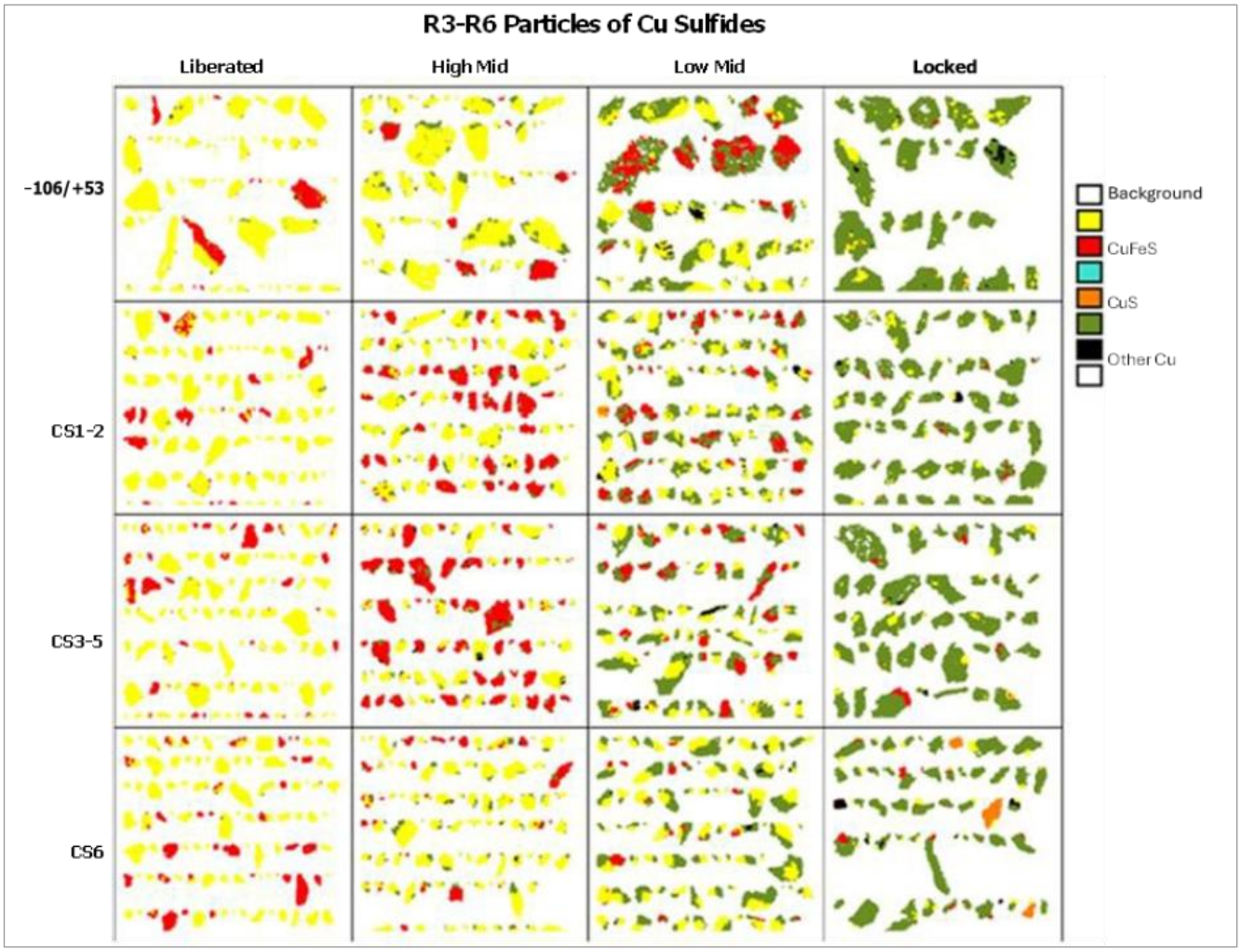


Source: Amec Foster Wheeler, 2018: from XPS data, 2014.

At the fine grind P80 of 38 µm, 27% of the copper sulphides remain unliberated. Almost half of these are in the very poor grade “locked” class and are generally unavailable to recover in flotation. If locked particles are recovered, they rarely survive the cleaning process and are rejected to tails at some point in the flow sheet.

QEMSCAN also generates particle mineral maps, and can group both minerals, and particles, to assist in visual examination. Figure 13.26 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 13.26 Combined copper sulphides liberation map – Rougher concentrates R3-R6



Source: XPS, 2015.

It is clear that even in the CS6 (cyclosizer cone 6 fraction, particle size about 4 µm) there is a large amount of the copper held in poorly liberated particles. The copper sulphide phases in the CS6 particles are typically 1–3 µm. This poor liberation of fine sulphides is a characteristic pervading the entire Kamoa mineralized zone and has driven the fine grinding component of the flow sheet development.

All particles in Figure 13.26 above have been floated or transported to the concentrate by entrainment with the froth water. All that is needed for a particle to float is a small exposure of copper sulphide at the surface and the “low Mid” and “Locked” particles in the image shows that this is generally the case.

The pervasive fine copper sulphides cause large amounts of attached silicates to be recovered in rougher flotation, and this leads to the high rougher mass pull values (20–40%) typical in the test programmes. At coarse grinds, such as 150 µm P80, large silicate particles invariably have exposed fine copper sulphides on the surface and are able to float.

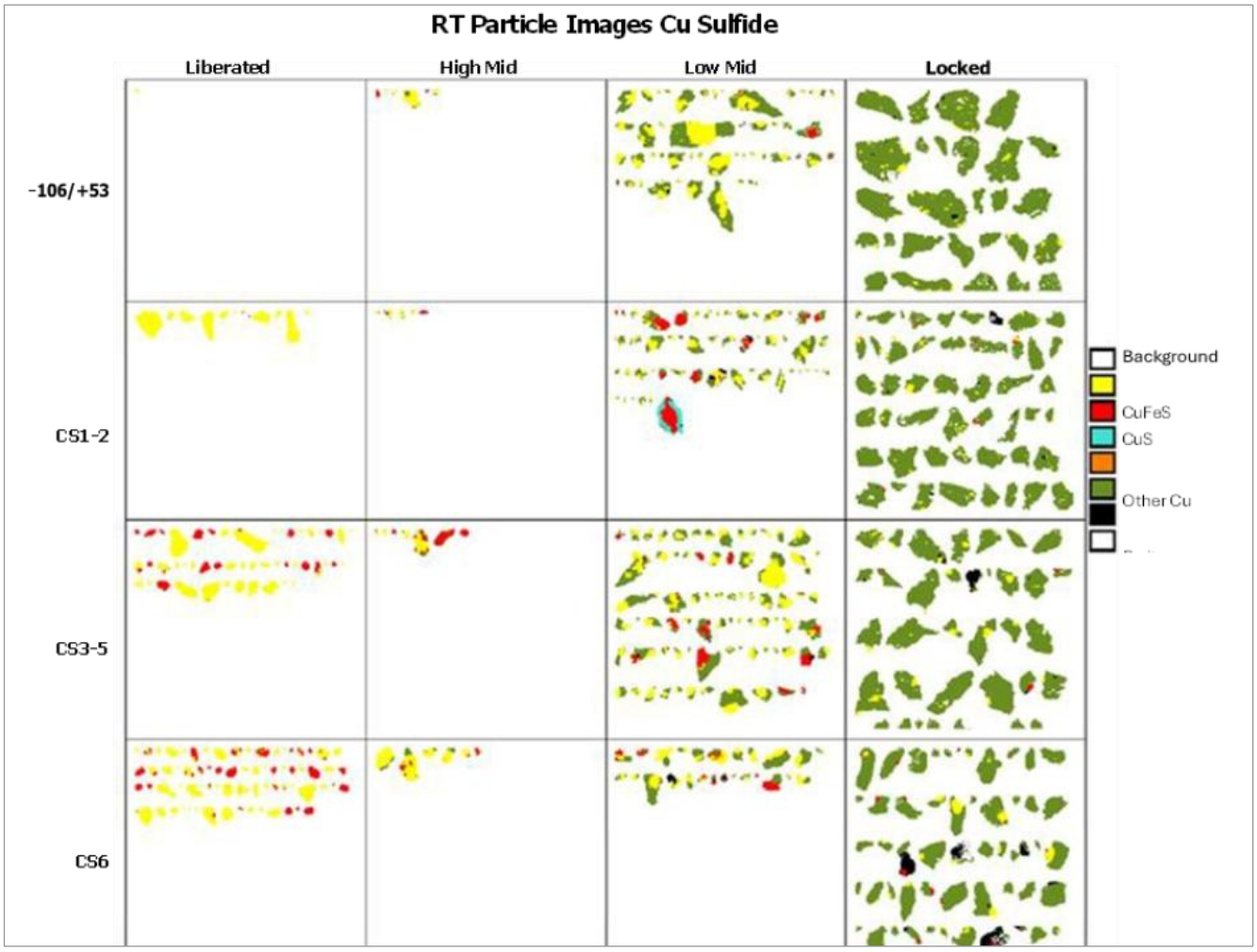
The fine sulphides also mean that regardless of the rougher flotation size it is necessary to regrind middlings material to ultra-fine sizes to achieve low silicate levels in final concentrates. Testing has shown the

concentrate quality to be sensitive to regrind P80, with 15 µm producing poor concentrates, and 10 µm generally producing acceptable concentrates.

Another notable aspect of Figure 13.26 above is the general absence of pyrite. It is only at the finest size that pyrite appears, and this indicates that composites or binary particles containing both pyrite and copper sulphides are scarce.

The major source of copper loss in flotation has been examined by QEMSCAN analysis of the rougher tailings. The liberation map for Rougher tails is shown in Figure 13.27.

Figure 13.27 Combined coper sulphides liberation map – Rougher tails



Source: XPS, 2015.

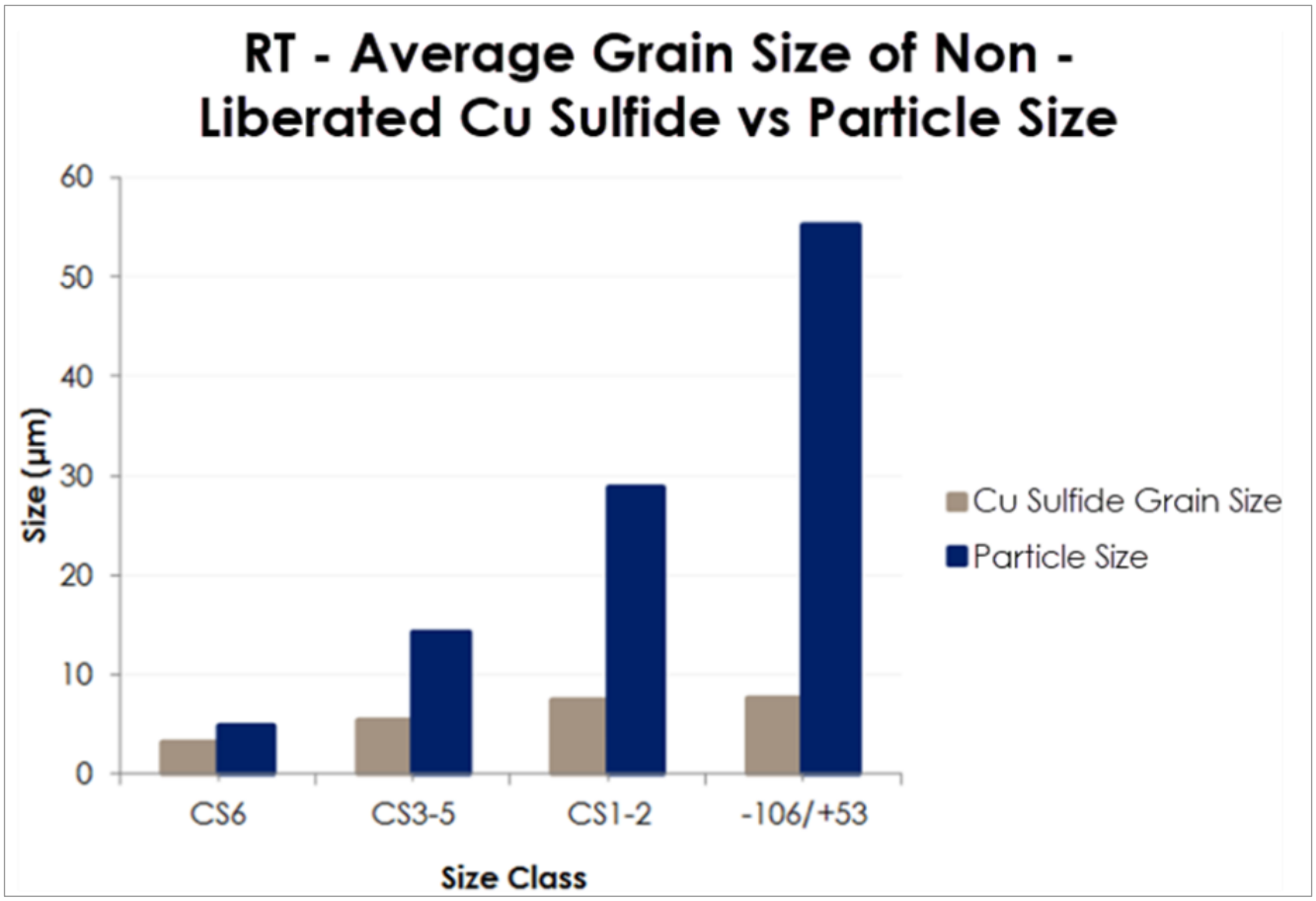
Although there are some fine liberated particles shown as being lost to rougher tailings, it is not possible, from this image alone, to determine how significant these few particles are in terms of copper recovery loss. Typically, the majority of lost copper will be in the Low-Mid, and the locked classes, simply because they represent the greatest mass proportion.

Many of the low mid particles may have floated with longer roughing time, but typically they report to tails because the surface of the sulphides is passivated or the actual amount of sulphide exposure is low (it

must be remembered that these images are particle cross-sections, and the real state of mineral exposure in three dimensions is unknown).

As can be seen in Figure 13.28, regardless of the size fraction, the lost copper sulphides are in phases that have average grain sizes of less than 10 µm.

Figure 13.28 Copper sulphide phase size in rougher tailings



Source: XPS, 2015.

The flotation test work has progressed to a point where recoveries in rougher flotation are typically above 90% and the material lost to tailings is dominated by ultra-fine locked copper sulphides. It has also progressed to the point where the need for ultrafine regrinding has been confirmed and high recoveries are being achieved at high concentrate grades.

13.2.5.7 Additional Kamoa Flotation variability campaign

Sample characterization

Kamoa Phase 3 samples were divided into Kamoa 1(K1) and Kamoa 2 (K2) zones. Three K1 and nine K2 samples were selected for variability test work. One Kansoko Sud sample was also selected for variability test work and for use in the final composite sample. Each of the samples was crushed to –2 mm (about 0.08 in) and homogenized. A 12 kg sub-sample was collected for the variability test work. The remaining samples were used to create K1 composite and K2 composite. The Life-of-Mine (LOM) composite, K3 was produced from a combination of K1, K2, and Kansoko Sud ores in a predetermined ratio as guided by the

mine plan at the time Thirteen drill core samples and three composite samples were tested, with copper head grades varying between 1.69-4.00% as reported in Table 13.17 and Table 13.18.

Table 13.17 Variability samples head assays

Sample ID	Average head assay (%)						
	Cu	Fe	S	SiO_2	MgO	Al_2O_3	CaO
DKMC_DD1725	3.22	5.86	2.76	59.0	3.13	12.3	0.68
DKMC_DD1727	3.19	5.68	2.03	60.3	2.52	12.4	0.18
DKMC_DD1728	1.67	4.93	1.51	63.7	3.28	11.7	1.50
DKMC_DD1708	3.31	5.80	2.00	61.9	2.10	14.90	0.26
DKMC_DD1710	2.05	8.28	3.81	58.1	3.82	13.95	0.16
DKMC_DD1711	1.90	3.25	1.80	62.4	1.21	14.90	0.21
DKMC_DD1712	2.78	4.94	2.69	62.2	1.71	16.00	0.70
DKMC_DD1714	2.18	5.21	1.68	66.1	3.30	13.65	0.16
DKMC_DD1716	3.38	4.61	1.35	61.1	2.77	14.00	1.14
DKMC_DD1718	2.93	7.70	3.52	59.9	2.79	13.90	0.52
DKMC_DD1720	3.10	4.36	0.77	66.2	2.64	12.60	0.49
DKMC_DD1724	4.00	4.44	1.00	66.9	2.29	11.75	0.46
DKMC_DD1733	3.03	6.58	2.26	59.2	4.02	13.4	0.41

Table 13.18 Composite head assays

Sample ID	Average head assay (%)						
	Cu	Fe	S	SiO_2	MgO	Al_2O_3	CaO
K1 composite	2.43	5.52	2.05	64.9	3.01	12.4	1.26
K2 composite	3.30	4.63	1.72	61.7	2.46	13.3	0.37
LOM comp (K3)	2.84	5.22	1.94	63.0	2.98	12.6	0.67

Sequential copper leach showed 0.09% and 0.52% acid soluble Cu for K1 and K2 respectively.

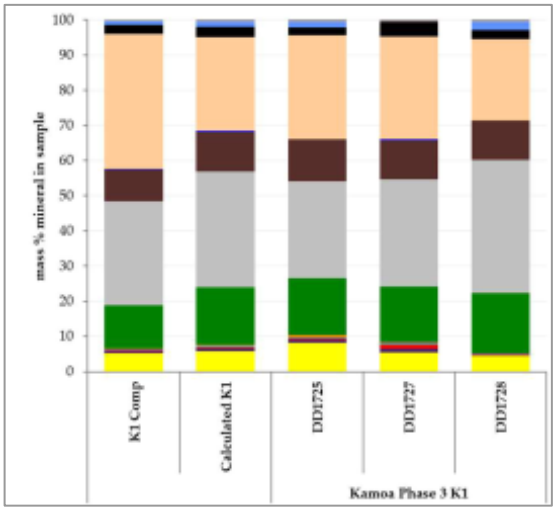
13.2.5.8 Mineralogy examination

K1 mineralogy

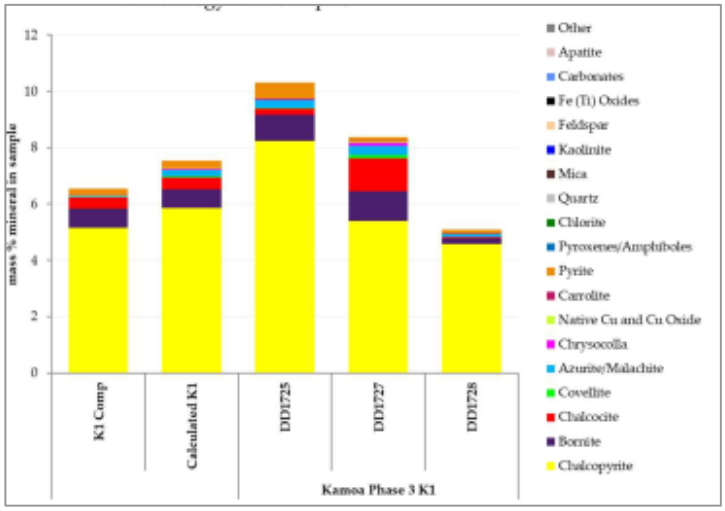
The K1 mineralogy is dominated by chalcopyrite, with lesser bornite and chalcocite.

Figure 13.29 K1 samples mineralogy

K1 samples bulk mineralogy



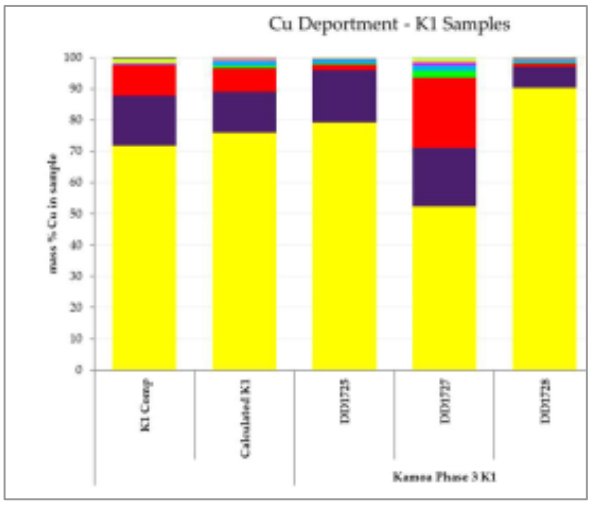
K1 samples sulphide plus oxide mineral



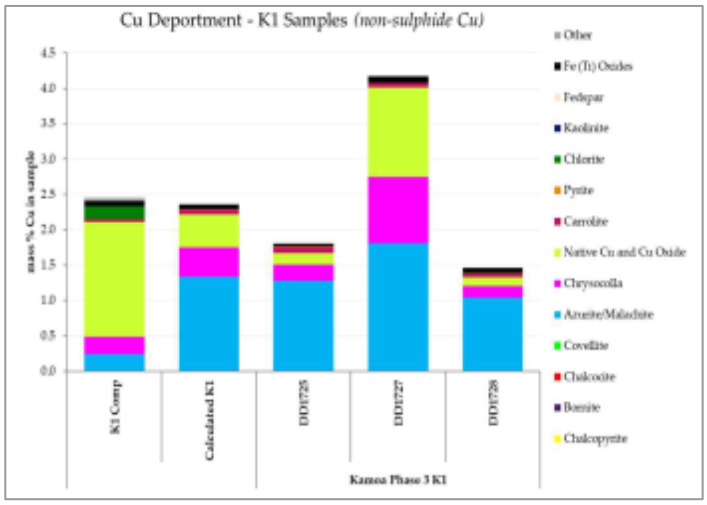
Source: XPS, 2024.

Figure 13.30 K1 samples Cu deportment

K1 samples Cu deportment



K1 samples non-sulphide Cu



Source: XPS, 2024.

K2 mineralogy

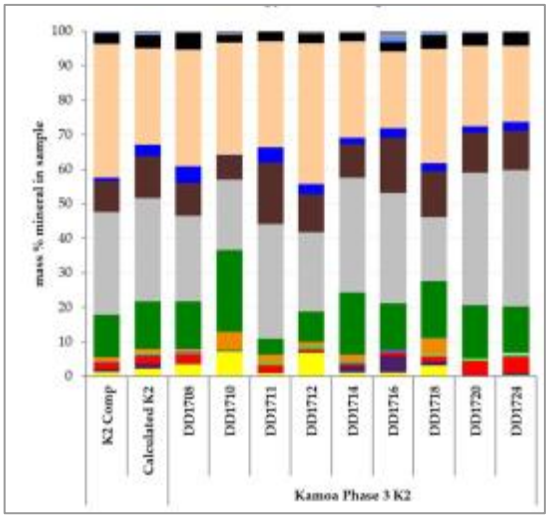
The K2 mineralogy shows a mixture of chalcopyrite, bornite or chalcocite rich components.

- DD1710 and DD1712 are mostly chalcopyrite.
- DD1714 and DD1716 are mostly bornite.
- DD1720 and DD1724 are mostly chalcocite.
- DD1708, DD1711 and DD1718 contain a mix of all three sulphide minerals.

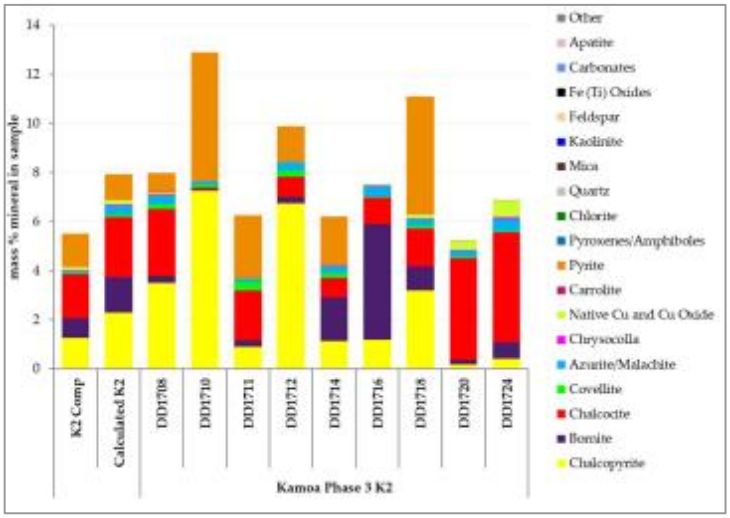
Non-sulphide copper is mostly azurite/malachite and native copper and copper oxide (abundant in DD1720 and DD1724).

Figure 13.31 K2 samples mineralogy

K2 samples bulk mineralogy



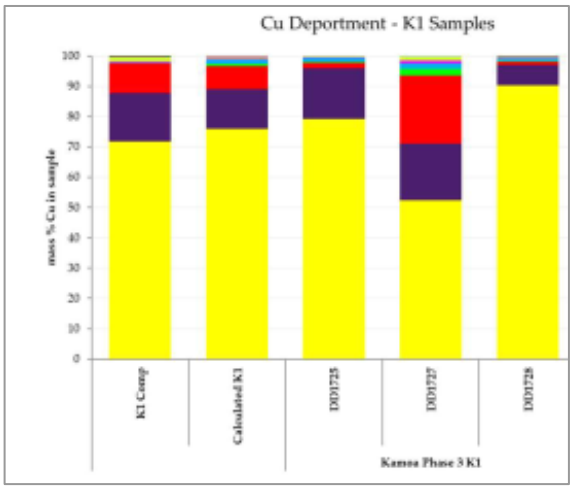
K2samples sulphide plus oxide mineral



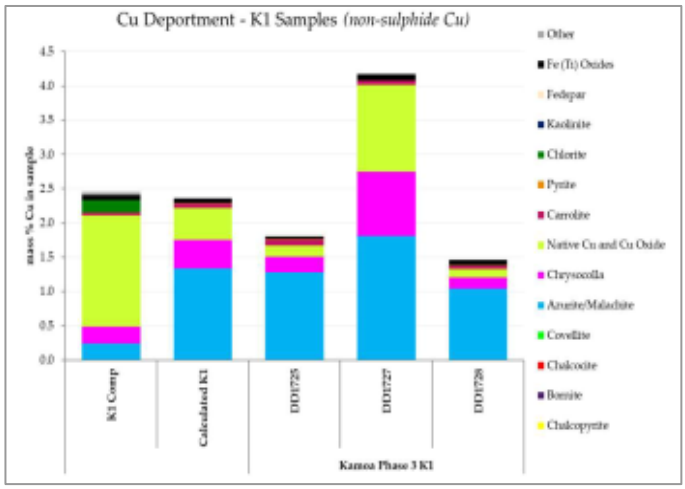
Source: XPS, 2024.

Figure 13.32 K2 samples Cu deportment

K1 samples Cu deportment



K1 samples non-sulphide Cu

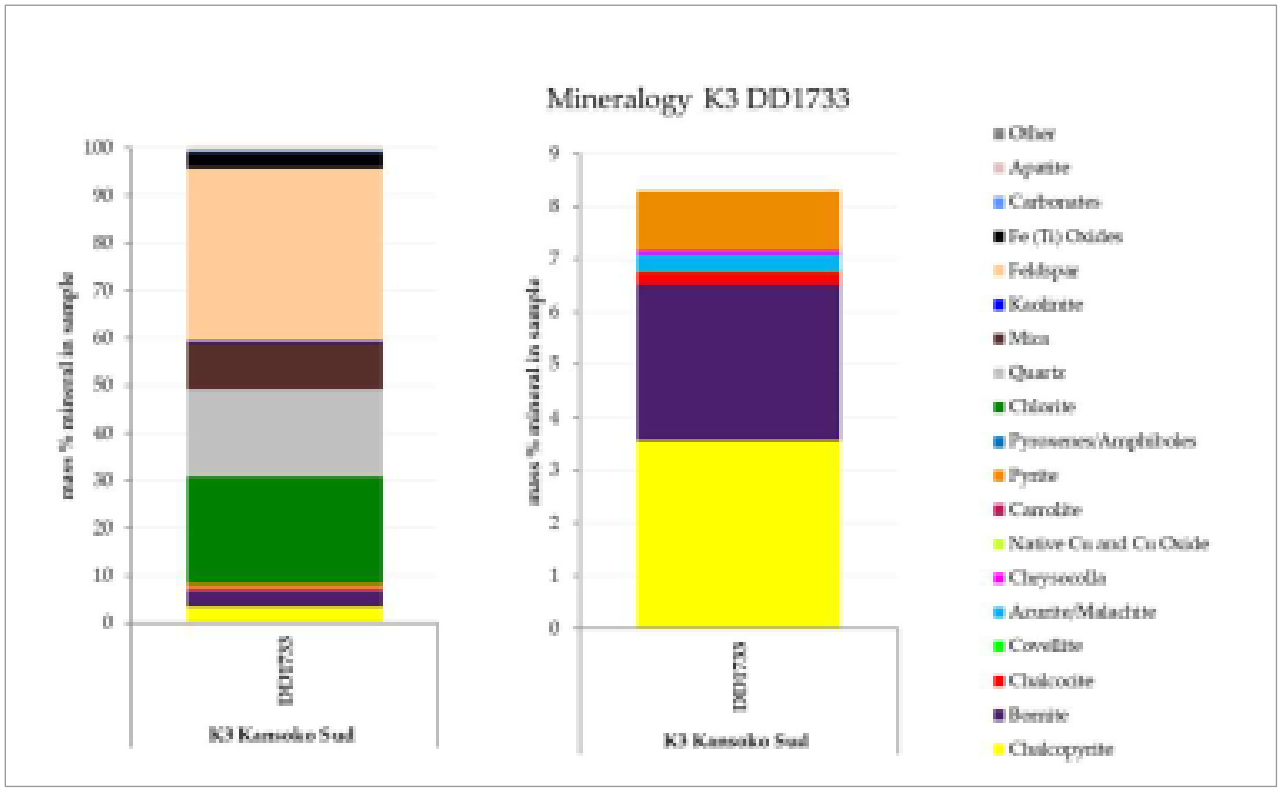


Source: XPS, 2024.

Kansoko Sud mineralogy

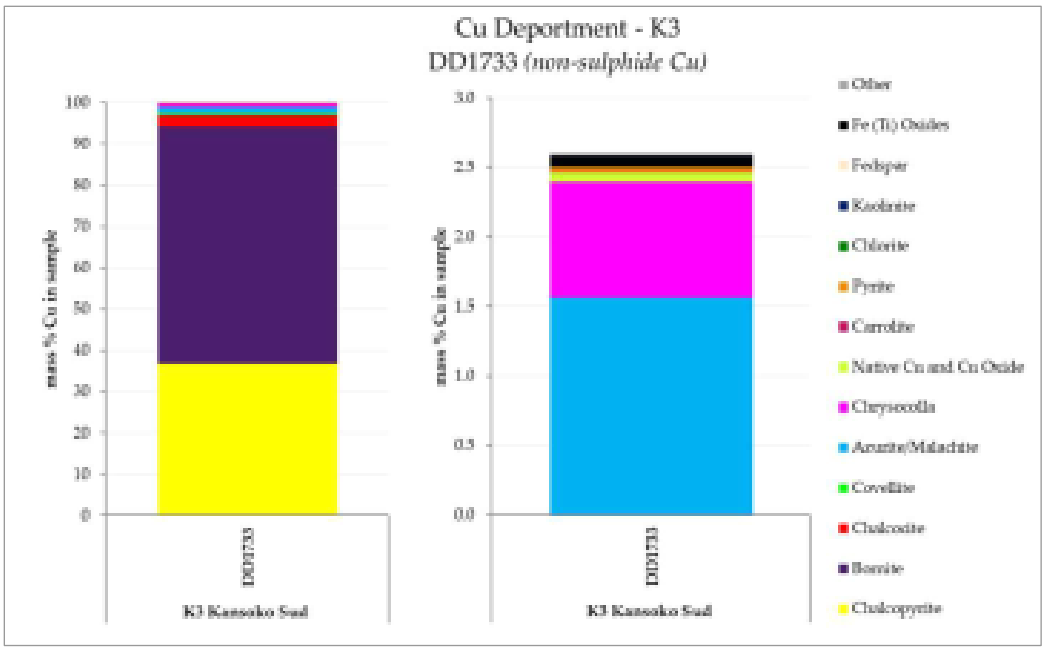
The sample is a mix of chalcopyrite and bornite with little chalcocite and trace azurite / malachite. The combine copper sulphide grain shows p80 of 60 µm. Figure 13.33 and Figure 13.40 show the mineralogy and copper deportment for K3 sample respectively.

Figure 13.33 K3 sample bulk mineralogy



Source: XPS, 2024.

Figure 13.34 K3 sample Cu deportment

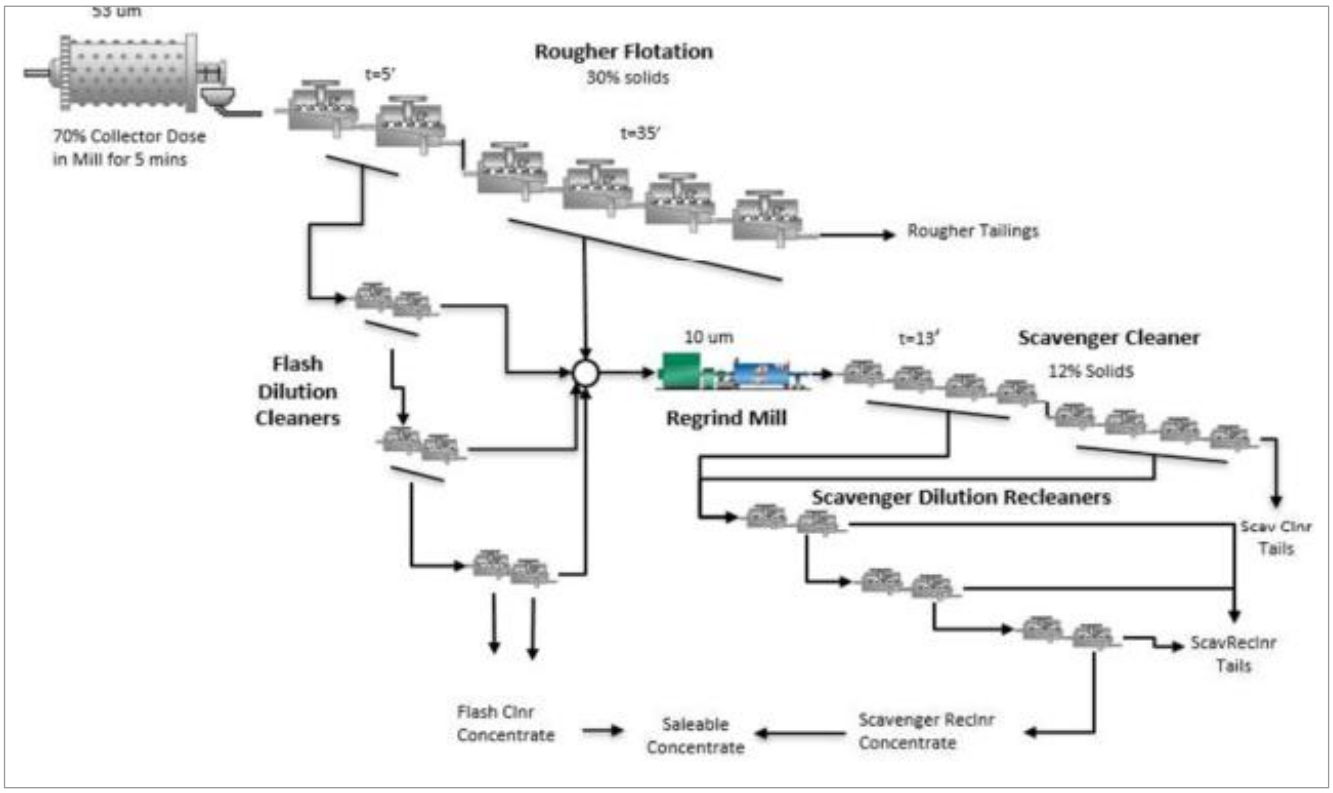


Source: XPS, 2024.

13.2.5.9 Kamoa Phase 3 flotation variability campaign

The IFS4a flow sheet was selected for the variability test work after preliminary flotation test work to confirm the flow sheet. The tests were to be conducted in duplicate, but after the first runs optimization of flotation conditions had to be done. Results for Run2 are referenced here. Figure 13.35 illustrates the Integrated Flowsheet 4a (IFS4a) used.

Figure 13.35 Kamoa variability flotation flow sheet (IFS4a)



Source: XPS, 2024.

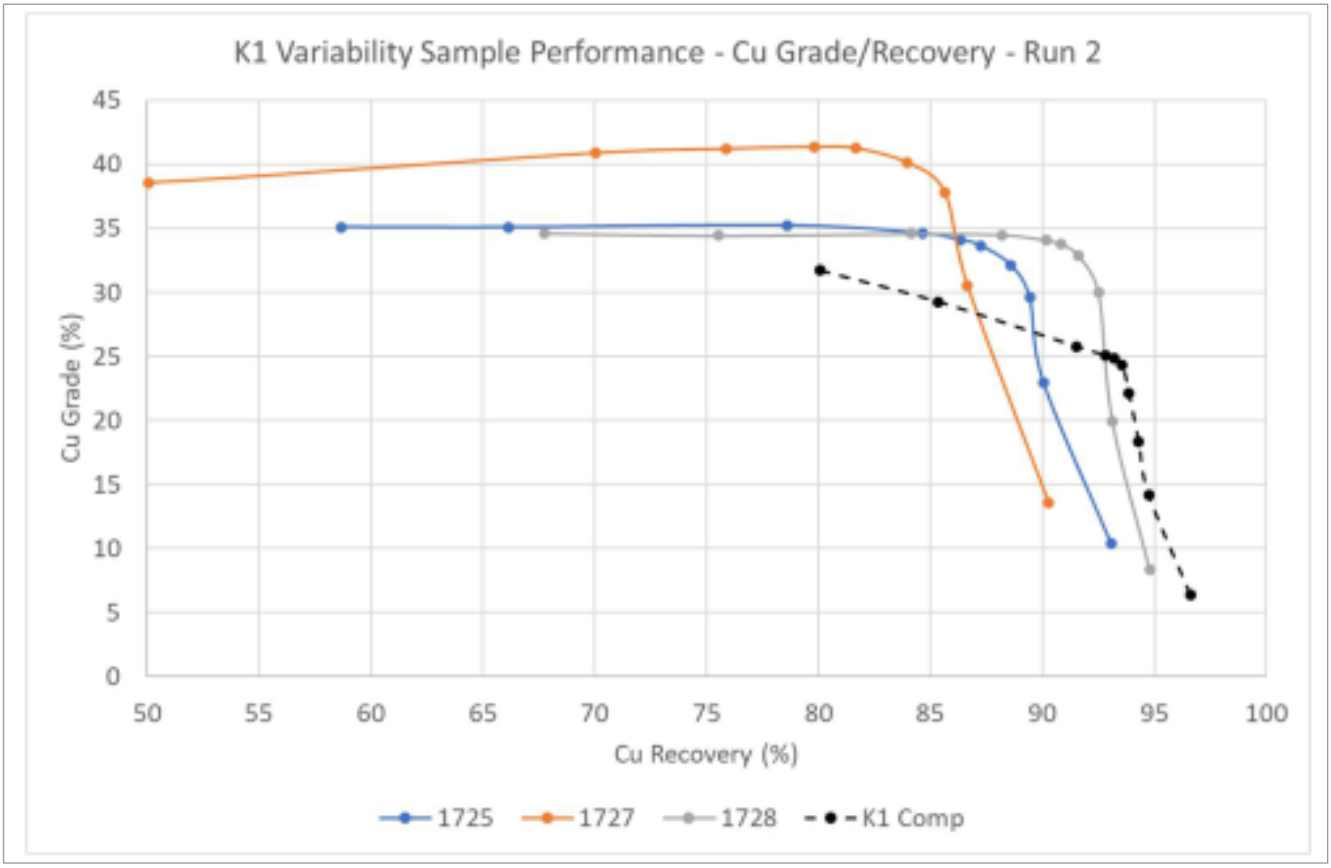
13.2.5.10 K1 variability flotation results

Table 13.19 shows the grade recovery relationship for the K1 samples. The combined cleaner concentrates range between 34% Cu and 41% Cu and recoveries between 82% and 91%. The composite sample gave a recovery of 94% at a copper concentrate grade of 24% for a mass pull of 9.5%. The Cu grade could be increased by reducing the mass pull with a reduction in recovery. The flotation results are summarized in Table 13.19.

Table 13.19 Summary of K1 samples and composite sample results

Samples	High Grade Clnr Conc			Scav Re-cleaner Conc			Combined Concentrate			
	% Cu	% SiO_2	% Cu Rec	% Cu	% SiO_2	% Cu Rec	% Mass	% Cu	% SiO_2	% Cu Rec
K1 composite	29.3	15.3	85.3	8.9	55.4	8.2	9.5	24.4	24.9	93.6
DD-1725	35.1	3.6	66.1	29.8	18.2	21.1	8.3	33.6	7.6	87.3
DD-1727	38.5	4.0	50.1	46.4	8.4	31.6	6.3	41.2	5.5	81.7
DD-1728	34.4	3.5	75.5	30.8	12.5	15.3	4.4	33.7	5.2	90.8

Figure 13.36 Grade recovery relationships for Kamoa variable test work samples



Source: XPS, 2024.

13.2.5.11 K2 variability flotation results

Results for K2 variability flotation are summarized in Table 13.20. The open cleaner circuit recoveries range from 77% to 91% and combined concentrate grade ranges from 19% Cu to 69% Cu. Sample DD1710 produced the lowest Cu concentrate grade. This is a sample that had elevated pyrite and was primarily chalcopyrite. The highest Cu concentrate grade was produced by samples DD1720 and DD1724, which were primarily chalcocite.

The locked cycle test with the LOM (K3) sample produced a concentrate with a grade of 36.7% Cu at a recovery of 84.6%.

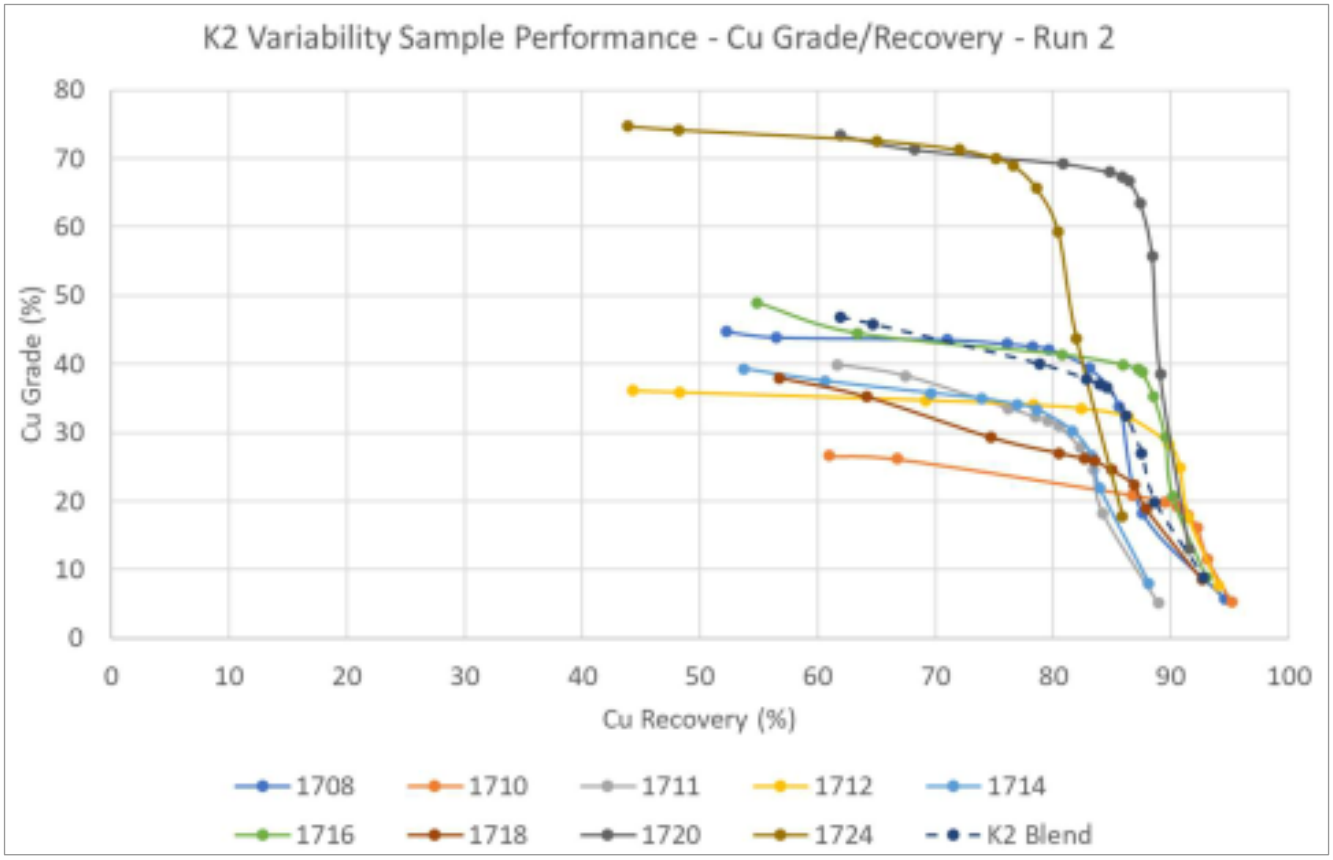
Table 13.20 Kamoa K2 variability test work results summary

Samples	High Grade Clnr Conc			Scav Re-cleaner Conc			Combined Concentrate			
	% Cu	% SiO_2	% Cu Rec	% Cu	% SiO_2	% Cu Rec	% Mass	% Cu	% SiO_2	% Cu Rec
K2 composite	45.8	12.6	64.7	22.2	34.4	19.8	7.7	36.7	21.1	84.6
DD-1708	43.9	7.2	56.5	38.4	14.8	23.1	6.4	42.1	9.6	79.6
DD-1710	26.2	2.9	66.8	10.6	18.8	24.2	10.2	18.8	10.4	90.9
DD-1711	38.3	8.0	67.4	15.5	24.1	13.1	5.1	30.9	13.2	80.5
DD-1712	35.9	3.2	48.3	28.6	13.0	38.1	7.5	32.3	8.0	86.3
DD-1714	37.5	10.6	60.6	24.4	17.3	18.0	5.2	33.4	12.7	78.6

Samples	High Grade Clnr Conc			Scav Re-cleaner Conc			Combined Concentrate			
	% Cu	% SiO_2	% Cu Rec	% Cu	% SiO_2	% Cu Rec	% Mass	% Cu	% SiO_2	% Cu Rec
DD-1716	57.6	3.7	46.3	51.5	13.4	36.4	5.2	33.4	12.7	78.6
DD-1718	35.2	10.9	64.2	13.8	19.2	19.3	9.3	26.0	14.5	83.5
DD-1720	71.3	5.5	68.3	53.8	16.3	18.2	4.2	66.8	8.3	86.4
DD-1724	74.2	3.1	48.2	61.5	11.9	28.4	4.6	68.9	6.8	76.6

The Cu Grade and recovery relationship is shown in Figure 13.37.

Figure 13.37 K2 samples grade recovery curves



Source: XPS, 2024.

13.2.5.12 LOM (K3) variability flotation test work results

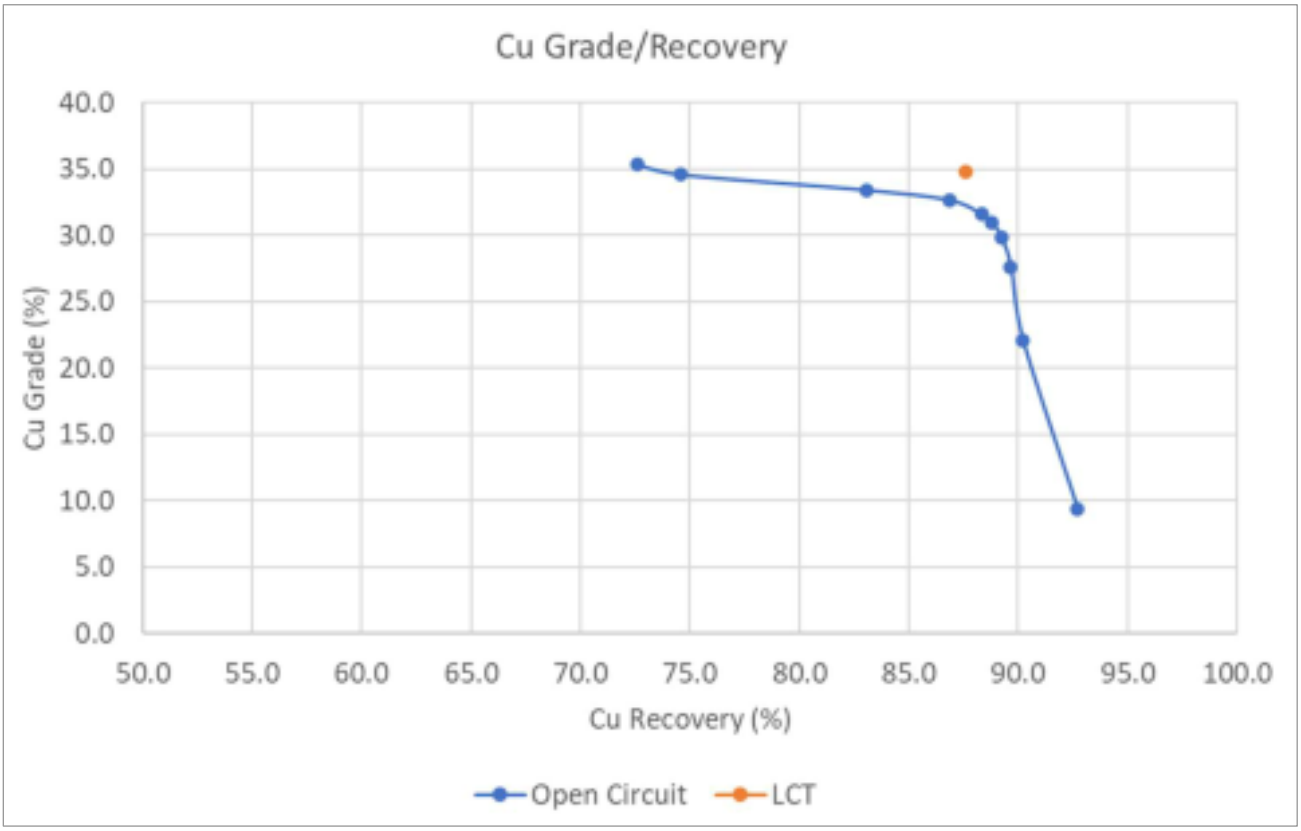
The locked cycle test work produced a recovery of 87.6% and a concentrate grade of 34.8% Cu at a mass pull of 7.0%. The results are summarized in Figure 13.23. The LCT was repeated for six cycles. The average of cycles 4-6 is reported.

Table 13.21 K3 composite LCT result

Product	Mass %	Assays (%)					
		Cu	Fe	S	SiO2	MgO	Al_2O_3
Calculated Head	100.00	2.79	5.69	1.97	63.2	3.10	13.3
Flash Clnr Conc	4.6	39.0	23.4	28.0	7.2	0.5	2.6
Scav Re-cleaner Conc	2.4	26.6	16.0	18.4	27.3	1.2	5.5
Scav cleaner tails	16.3	0.68	5.04	0.44	64.2	3.4	15.5
Scav float tails	76.7	0.31	4.43	0.21	67.5	3.2	13.7
Total cleaner concentrate	7.0	34.8	20.9	24.8	14.0	0.8	3.58
		Distribution (%)					
Calculated Head	-	100.0	100.0	100.0	100.0	100.0	100.0
Flash Clnr Conc	-	64.9	19.1	65.8	0.5	0.8	0.9
Scav Re-cleaner Conc	-	22.7	6.7	22.2	1.0	0.9	1.0
Scav cleaner tails	-	4.0	14.5	3.7	16.6	17.9	19.0
Scav float tails	-	8.4	59.7	8.3	81.9	80.4	79.1
Total cleaner concentrate	-	87.6	25.8	88.0	1.6	1.7	1.9

The open circuit flotation grade recovery curve for sample K3 is shown in Figure 13.38. The locked cycle test result is superimposed on the same graph.

Figure 13.38 Sample K3 grade recovery curve



Source: XPS, 2024.

13.2.6 Metallurgical test work on Kakula Resource

The initial metallurgical test work, 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 focused on flow sheet optimization as part of the Kakula project PFS.

The PFS test work campaign (2017–2018) consisted of the following:

- Mineralogy and sample characterization on a mill feed, and a final concentrate sample, conducted by XPS.
- Comminution testing, conducted by Mintek.
- Flotation flow sheet optimization and preliminary variability testing, conducted by XPS.
- HPGR scoping and pilot plant testing, conducted by ThyssenKrupp, South Africa.
- Concentrate thickening and filtration test work, conducted by Outotec, Canada.
- Tailing thickening and filtration test work, conducted by SGS, Canada.
- Bulk material flow test work, completed by GreenTechnical in South Africa.

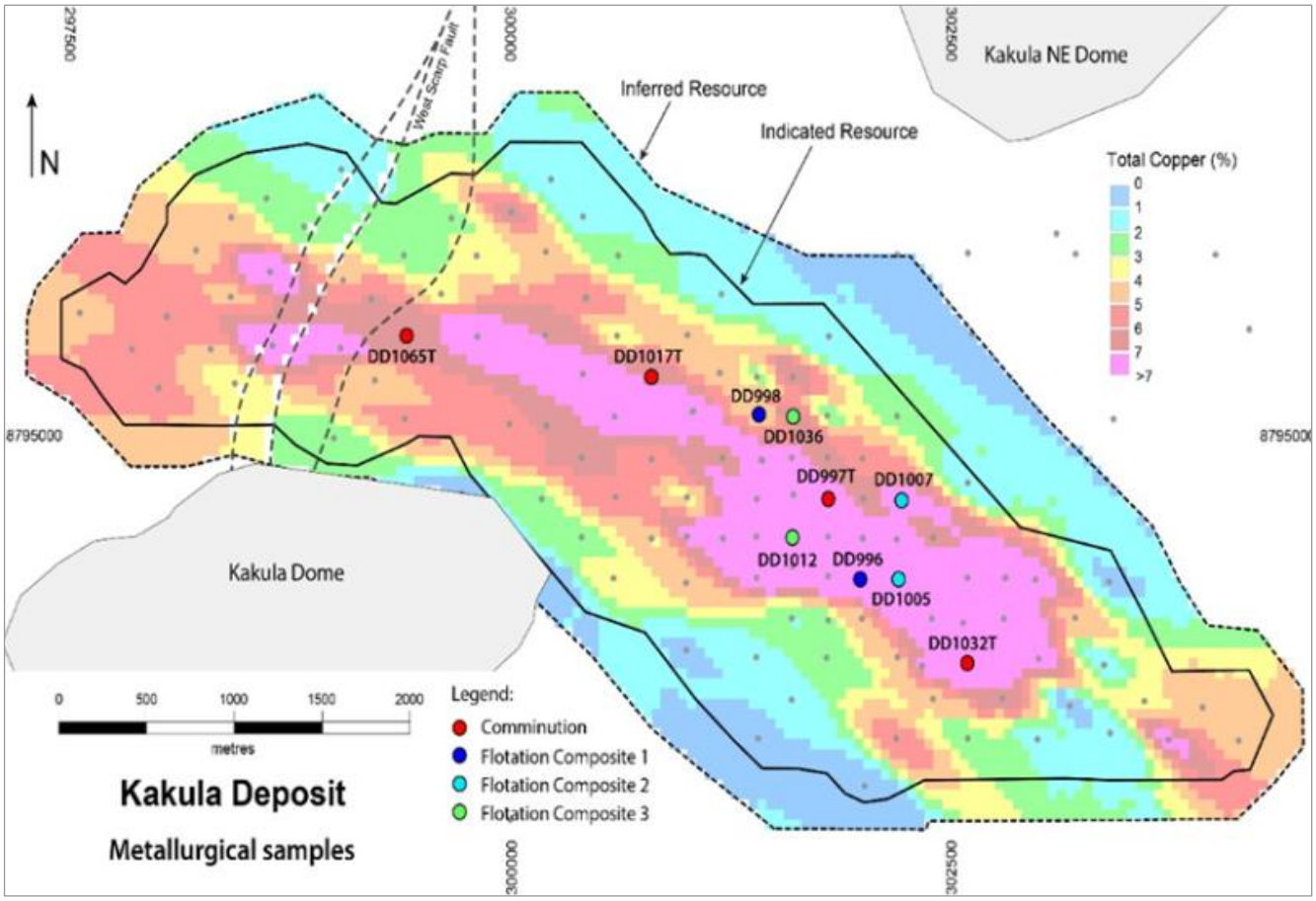
Further test work was initiated in March 2019 as part of the feasibility study and consisted of:

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

13.2.7 Kakula metallurgical sample locations and descriptions

Refer to Figure 13.39 for an illustration of the positions of each of the drill cores tested during the preliminary and PSF test work campaigns.

Figure 13.39 Drillhole location map of Kakula metallurgical samples



Source: KCSA 2023

13.2.7.1 Preliminary flotation sample

Preliminary flotation test work, 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 test work campaign.

Head analyses were conducted in triplicates on each of the above flotation composite samples. Refer to Table 13.22 below for a summary of the head analysis results for the various flotation samples.

Table 13.22 Kakula preliminary flotation samples head analysis

Sample	Cu (%)	S (%)	SiO2 (%)	Fe (%)	Al2O3 (%)	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**

13.2.7.2 Kakula FFS comminution sample

Four PQ drillhole samples (DD1065T, DD997T, DD1017T, and DD1032T) were selected for comminution testing. The different lithologies (footwall (FW), diamictite (SDT) and siltstone (SSL)), per hole were 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.

13.2.7.3 Kakula comminution test work

Mintek completed the comminution parameter variability test work in 2018. The scope of work included:

- Uni axial compressive strength (UCS)
- Bond CWi and drop weight tests (DWi)
- SAG mill comminution (SMC)
- Bond abrasion index (Ai)
- BRWi
- BBWi at 75 µm closing screen sizes

The results of the above tests are summarized in Table 13.23 and Table 13.24.

Table 13.23 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 13.24 Kakula PFS SMC parameters summary

Sample ID	Mia (kWh/t)	Mih (kWh/t)	Mic (kWh/t)	ta	A	b	Axb
DD997T SSL	29.7	25.0	12.9	0.21	75.4	0.31	23.4
DD1065T SDT	30.6	25.7	13.3	0.21	80.8	0.28	22.6
DD1047W1 SDT	24.8	19.8	10.2	0.26	72.9	0.40	29.2
DD1084W1 SDT	25.4	20.3	10.5	0.26	62.1	0.46	28.6
DD1021W2 SSL	26.2	21.3	11.0	0.24	73.5	0.37	27.2
DD1047W1 SSL	24.9	20.0	10.3	0.26	72.9	0.40	29.2
DD1061W1 SSL	27.5	22.9	11.8	0.22	80.2	0.32	25.7
DD1070W1 SSL	27.2	22.3	11.6	0.23	80.3	0.32	25.7
DD1145W2 SSL	17.1	12.4	6.4	0.42	59.6	0.78	46.5
FW SDT 1	21.2	16.2	8.4	0.33	68.0	0.53	36.0
FW SST 2	**22.1**	**16.6**	**8.6**	**0.35**	**75.6**	**0.46**	**34.8**

Initial PFS CWi testing indicated that the Kakula PFS material was soft with regards to crushing energy requirements – however, observations made during the testing noted the presence of pre-existing cracks in the core which was most likely responsible for the low CWi values measured. During the latest testing, the CWi values averaged 11.3 kWh/t for the diamictite samples, and 13.4 kWh/t for the siltstone samples. The diamictite samples compared well to the earlier tested samples, however, the siltstone CWi increased

from 10.6 kWh/t measured earlier. DWi values averaged 10.0 kWh/m^3 for the diamictite samples, and 10.1 kWh/m^3 for the siltstone samples.

The Ai results generally demonstrated low abrasion tendencies for the Kakula material. The Ai measurements averaged 0.02 g for the diamictite samples, and 0.04 g for the siltstone samples.

The BRWi results grouped the Kakula material in the hard–very-hard classes, while the BBWi testing grouped all the samples in the very hard class. The variability samples indicated that the BBWi values averaged 18.2 kWh/t for the diamictite samples, and 17.5 kWh/t for the siltstone samples.

SMC testing also classified the samples tested as very hard, indicating that the Kamoa and Kakula material was highly competent and not amenable to Semi and/or Fully Autogenous Milling. The Axb values ranges from 22.6–46.5 (average 29.9). The maximum values are significantly higher compared to the Kamoa Phase 6 samples (17–28).

The Kakula PFS samples tested had similar competency compared to the Kamoa Phase 6 material.

13.2.7.4 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 test work 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 summarized in Table 13.25.

Table 13.25 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 test work.

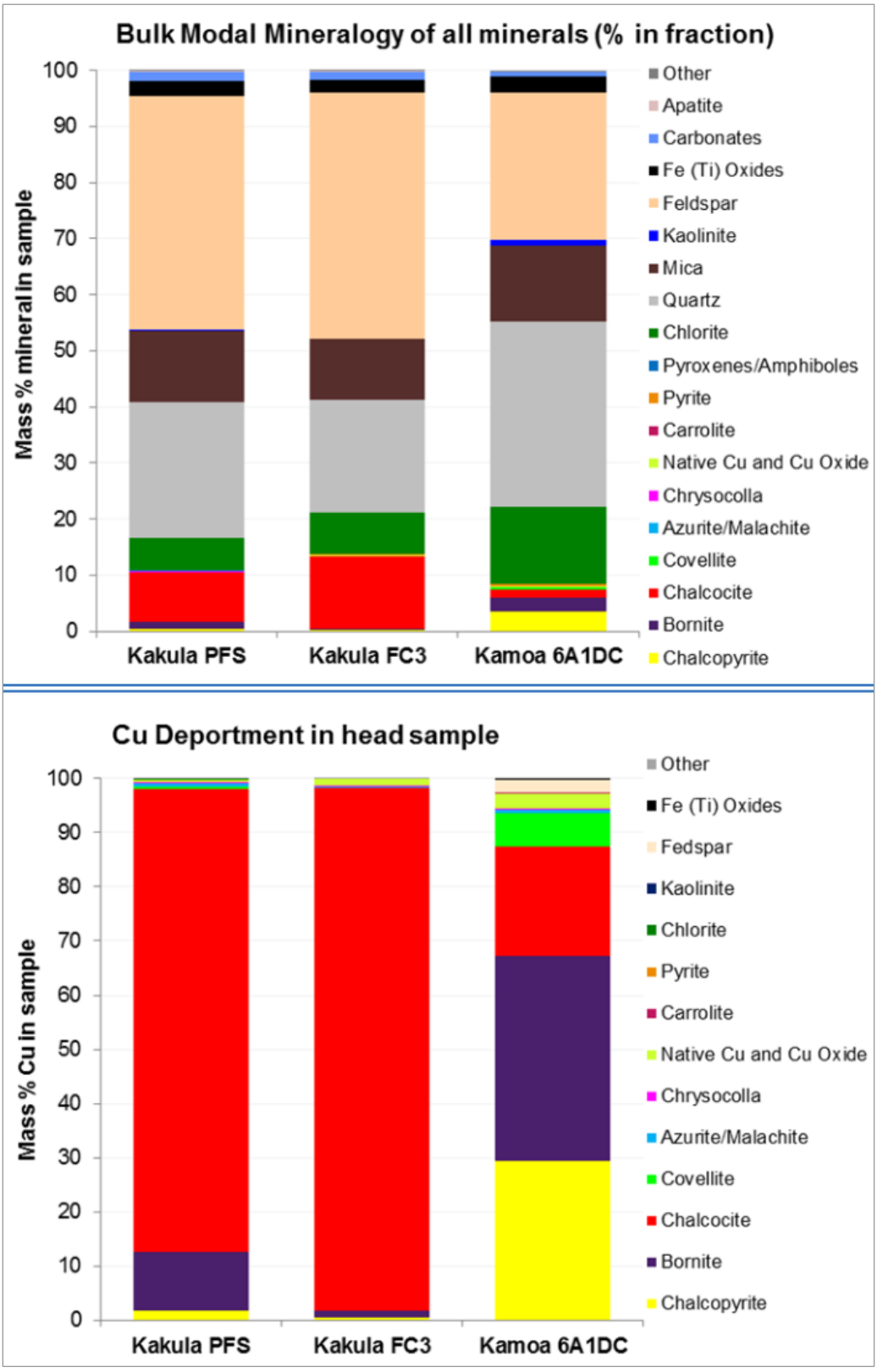
13.2.8 Mineralogical studies

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 was compared to the Kamoa Phase 6 development composite sample (Kamoa 6A1DC).

Further mineralogical investigations were conducted by XPS during 2017–2018, as part of the PFS flow sheet development. QEMSCAN was used on the Kakula PFS flotation master composite sample (Kakula PFS) to determine the bulk modal mineralogy, average grain size, liberation, and level of locking of sulphide particles in each sample.

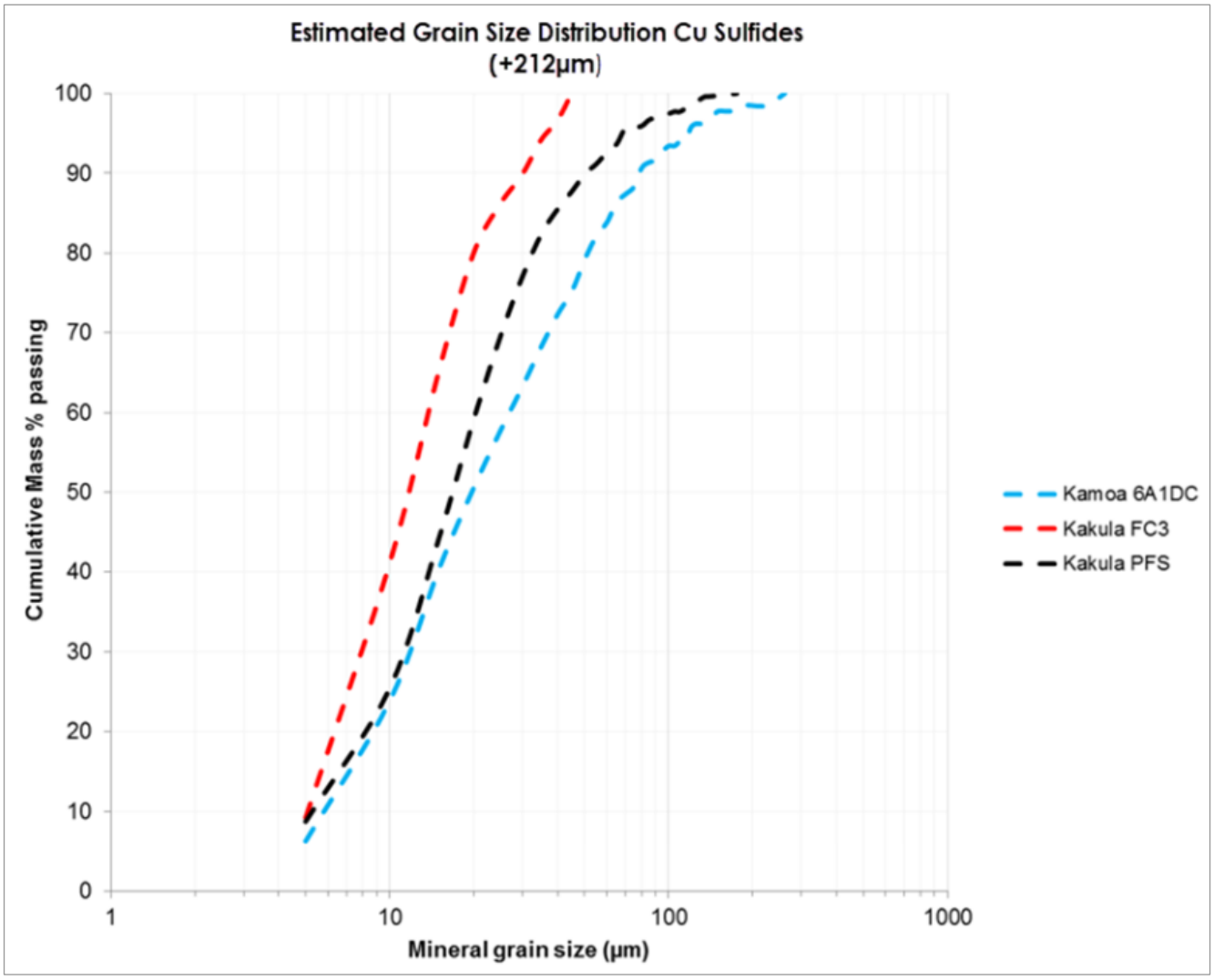
Figure 13.40 below summarizes the results from the Kakula PFS sample bulk modal analysis, as compared to the Kamoa 6A1DC sample and the Kakula FC3 sample. Refer to Figure 13.41 for a comparison of the combined Cu sulphide grain size distributions between Kamoa 6A1DC and Kakula FC3 samples.

Figure 13.40 Kamoa 6A1DC, Kakula FC3 and Kakula PFS samples mineralogy



Source: Ivanhoe, 2018.

Figure 13.41 Cu sulphide grain size distribution comparison between Kamoa and Kakula



Source: Ivanhoe, 2018.

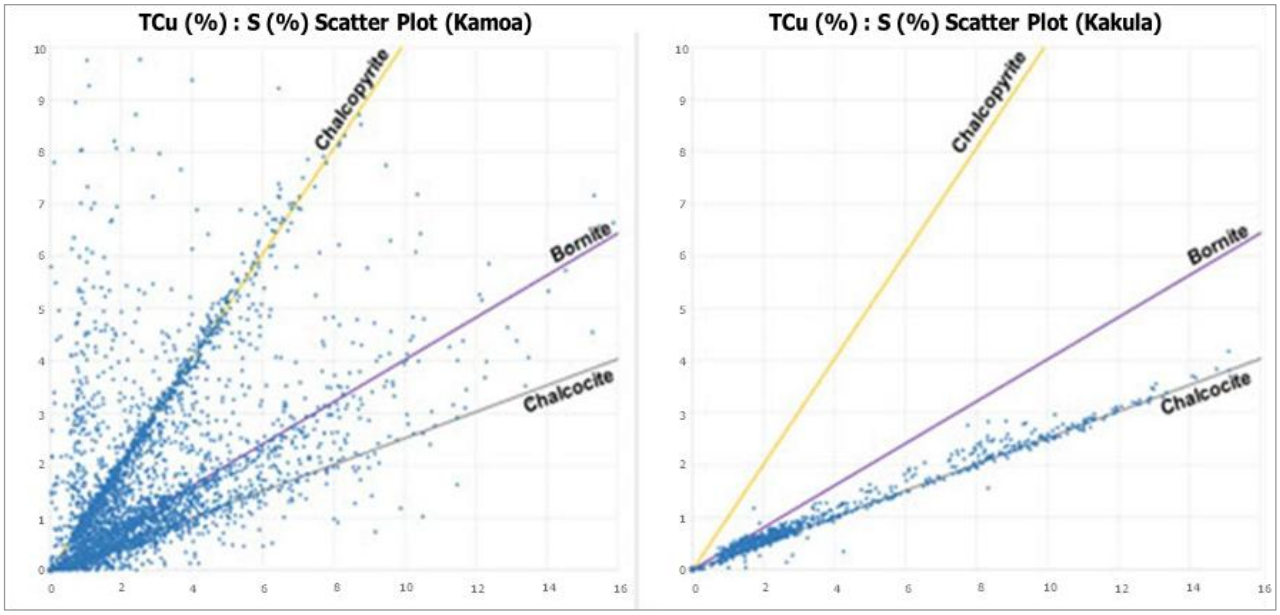
13.2.9 Dominance of chalcocite

The following was noted:

- The main Cu sulphide mineral in the Kakula samples was chalcocite, with minor amounts of bornite and covellite. Trace amounts of chalcopyrite were 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 sulphide was 33 µm, which was slightly coarser than the Kamoa 6A1DC sample (20 µm). The Kakula FC3 however, had a finer grain size of 9 µm, showing variation in the Kakula material grain sizes.

- Although Liberation data at the product size of 80% –220 µm showed that the total of the "liberated plus free" classes is effectively equal for each sample, at approximately 45%, more mineral occurs in the "free" class for the Kakula FC3 sample. There are major differences at the "locked" end of the comparison with the Kakula FC3 sample. Having approximately half the locked Cu of the Kamoa 6A1DC sample.
- The average grain size of the Cu sulphide minerals in the Kakula PFS composite sample was finer than the Kamoa 6A1DC sample at 12 µm, which was consistent with the Kakula FC3 sample (10 µm). Approximately 25% of the Cu sulphide minerals' mass occur in the sub 10 µm ranges, while roughly 8% occurs in the sub 5 µm range.
- Chalcocite is a high-tenor mineral that is opaque and dark grey to black with a metallic luster. Owing to its very high percentage of contained copper by weight and its capacity to produce a clean, high-grade concentrate, chalcocite is an asset as a dominant copper mineral. Unlike Kamoa, the Kakula deposit has very low bornite, chalcopyrite or other sulphide minerals as seen in Figure 13.42.

Figure 13.42 Comparison of Cu:S between Kamoa and Kakula mineralization



Source: Ivanhoe, 2016.

13.2.10 Kakula preliminary flotation test work

The initial flotation test work 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.
- Duplicate tests on DD996, DD998, and flotation composite 1 using the IFS4a flow sheet as developed during the Kamoa test work 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 was 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.
- 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 summarized in Table 13.26 below.

Table 13.26 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 optimization tests. Refer to Table 13.27 for a summary of the results obtained.

Table 13.27 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
Optimized flow sheet 1	12.4	86.2	56.1	11.4	15.5	3.7	3.2	<0.01
Optimized 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 optimized 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 13.28 for a summary of the results obtained.

Table 13.28 Flotation composite 3 flotation performance by XPS

Test Reference	Mass pull (%)	Recovery (% Cu)	Final Concentrate Grades (%)					
			Cu	SiO2	S	Fe	Al2O3	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.

13.2.11 Kakula PFS flotation flow sheet development test work

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 optimize 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 optimization of pulp densities, reagents, and additions, regrind circuit, and low entrainment cleaning.

13.2.11.1 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 13.2.7).

13.2.12 Kakula flow sheet development and optimization

Several 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.

- Optimization of high-grade cleaner circuit kinetics by varying collector dosages and flotation residence times.
- Optimization of scavenger cleaner circuit kinetics by varying collector dosages, phased collector additions, and flotation residence times.
- Optimization 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.
- Optimization of the final product grades by using low entrainment cleaning.

The following was noted:

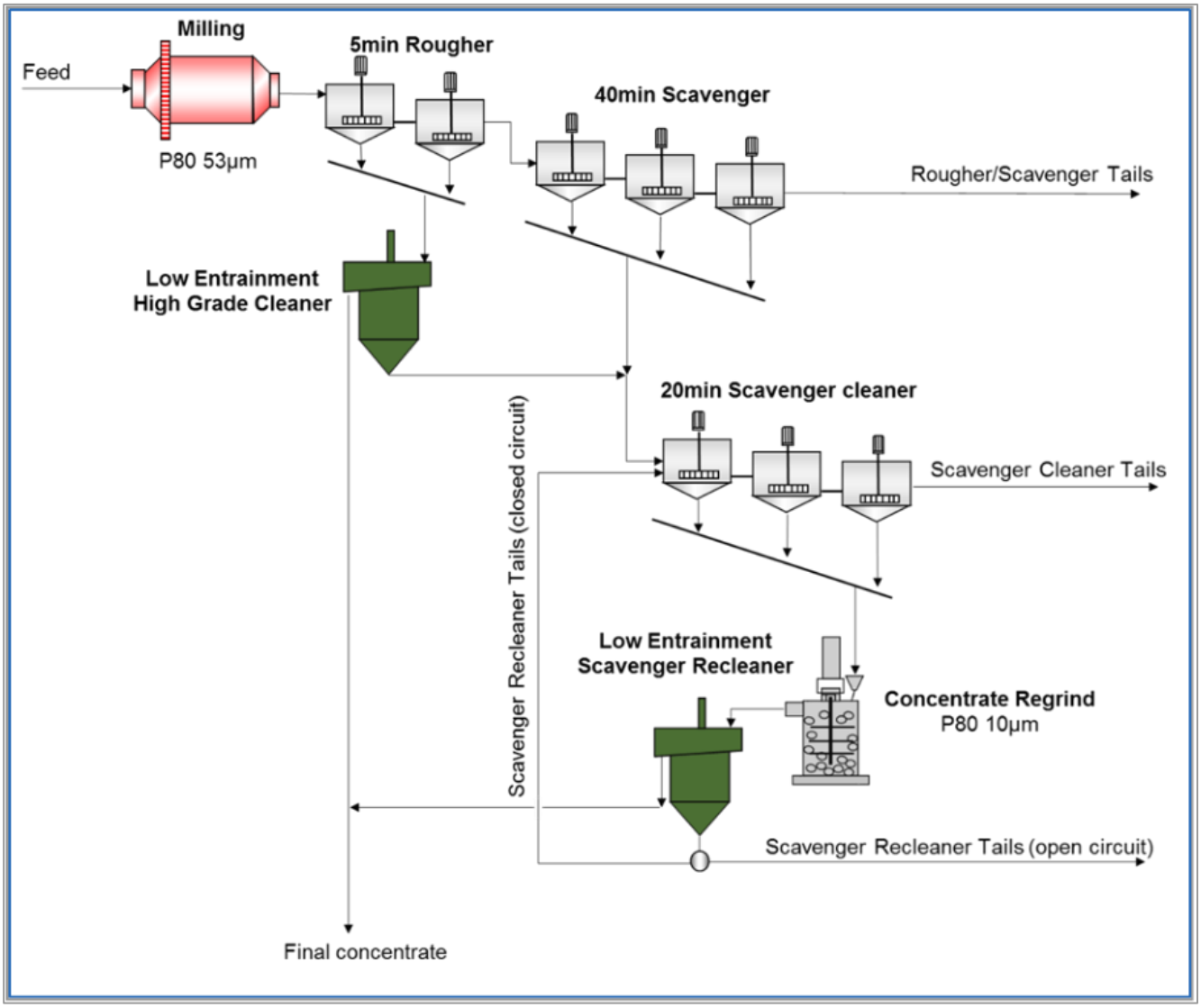
- Increasing the rougher pulp density to 35% did not impact on Cu recovery but did result in high silica recovery to the high-grade circuit. This can be managed by further cleaning.
- Modifications to the cleaner circuit did not result in any significant changes in overall recovery and only ended up 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.

13.2.13 Kakula PFS flow sheet

Figure 13.43 illustrates the final Kakula PFS flow sheet. A summary of the flow sheet conditions is given in Table 13.29.

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 13.43 Kakula PFS flow sheet



Source: Kamoa Copper SA, 2017

Table 13.29 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

Table 13.30 Kakula PFS flotation parameters

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 P80	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	–	–	–

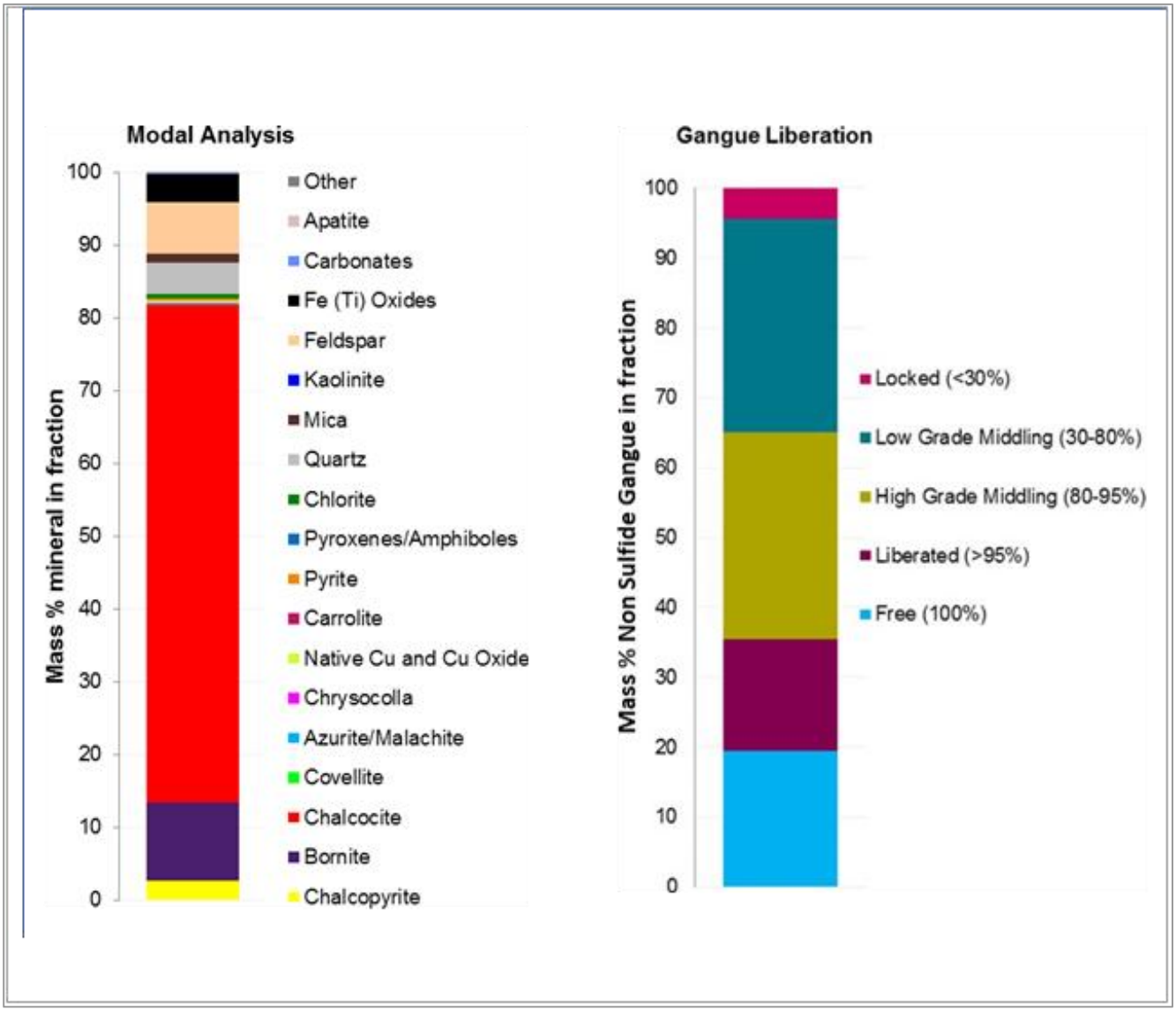
13.2.14 Flotation products mineralogy

Mineralogy was conducted on a single rougher tailings sample, to determine the major cause of Cu losses to this stream. The Cu deportment indicated 86% of Cu to sulphides, of which the majority was chalcocite. This indicated that poor liberation was at fault for these Cu losses, rather than mode of occurrence. The average grain size of the Cu minerals in the rougher tailings was 3–5 µm. Almost 92% of the Cu sulphide minerals in the rougher tailings was locked – none of the Cu sulphide minerals in the rougher tailings were noted as being contained within the free or liberated classes.

Mineralogy was also conducted on a bulk concentrate sample, produced during the development campaign. Refer to Figure 13.44 for an illustration of the bulk modal and gangue liberation information. The modal analysis showed that 81.8% of all minerals occurred as Cu sulphides of which 86% of the Cu as sulphides occurred as chalcocite, 11% as bornite and 1.5% as chalcopyrite. The main gangue minerals in the concentrate were feldspar, quartz, and Fe (Ti) oxides. Approximately 35% of the gangue that reported to the concentrate was in the free and liberated liberation classes.

Refer to Table 13.31 for the results of the full chemical analysis conducted on the Kakula PFS composite sample final concentrate product.

Figure 13.44 Kakula PFS concentrate modal analysis and gangue liberation



Source: Ivanhoe 2018

Table 13.31 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

Element	Units	High-grade Concentrate	Recleaner Concentrate	Combined Concentrate
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

Element	Units	High-grade Concentrate	Recleaner Concentrate	Combined Concentrate
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

13.2.15 Flotation variability campaign

13.2.15.1 Sample characterization

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 summarized in Table 13.32.

Table 13.32 Kakula preliminary flotation variability samples head grade analysis

Sample	Cu (%)	S (%)	SiO_2 (%)	Fe (%)	Al_2O_3 (%)	CaO (%)	MgO(%)	As (%)
DKMC_DD1017TW1	8.03	2.13	52.60	4.50	13.17	1.03	3.96	0.001
DKMC_DD1020W1	9.24	2.22	55.33	5.02	14.00	1.04	3.56	0.002
DKMC_DD1029W1	2.64	0.65	56.10	4.84	13.50	2.27	4.20	0.001
DKMC_DD1032W1	4.96	1.20	55.53	4.87	12.77	1.86	4.75	0.001
DKMC_DD1043W1	5.97	1.48	56.73	4.87	13.90	1.84	4.16	0.003
DKMC_DD1065W1	5.58	2.20	55.57	5.35	14.47	1.25	3.69	0.001
DKMC_DD1075W1	5.34	1.04	56.17	4.62	13.73	0.67	3.77	0.001
DKMC_DD1081W1	5.29	1.25	54.43	5.32	13.37	1.89	4.26	0.002
DKMC_DD1112W1	5.17	1.47	54.50	5.79	13.07	1.55	4.13	0.001
DKMC_DD997TW1	6.66	1.61	52.20	4.87	13.30	1.78	3.95	0.001

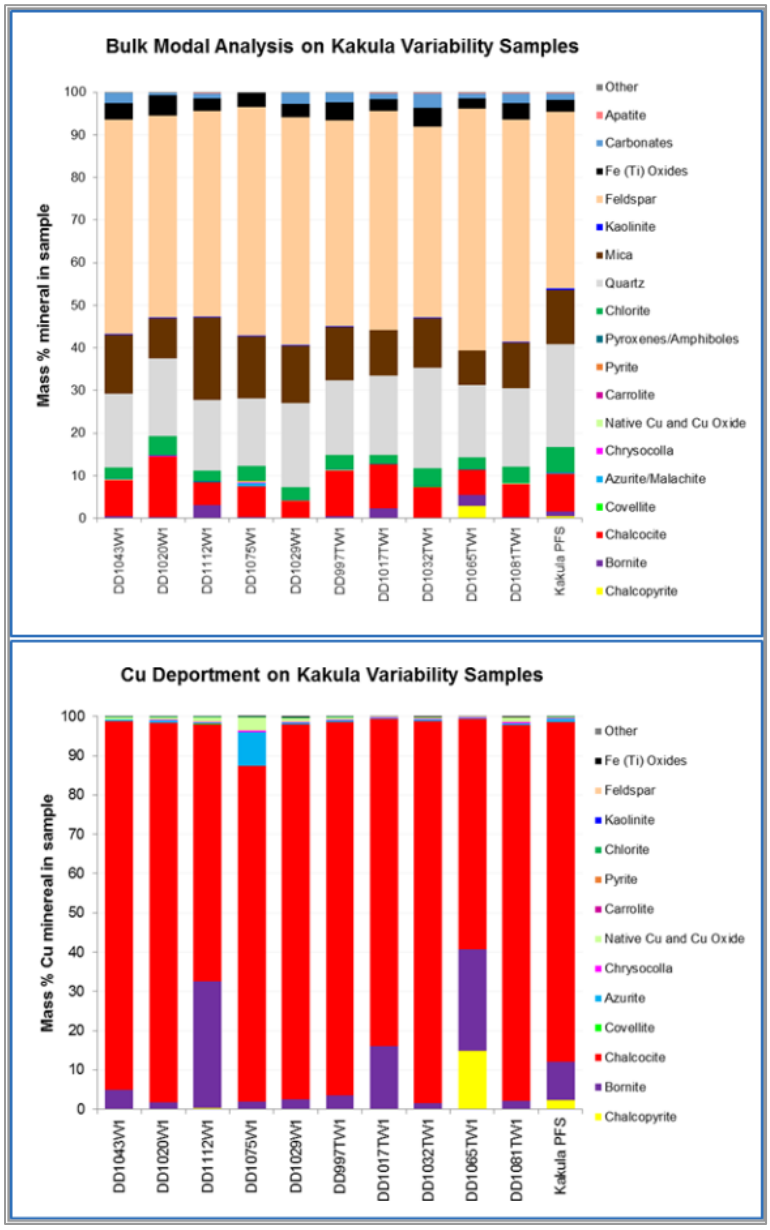
The following was noted from the head grade analysis:

- The samples tested varied from 2.6% Cu to 9.2% Cu, with sulphur grades generally increasing with increasing Cu grades.
- Fe, MgO, and Al_2O_3 values were relatively constant over the range of samples, averaging 5.0%, 4.0% and 13.5% respectively.
- The highest As value measure was 0.003% for sample DD1043W1, with the majority of the samples reported as below the instrument detection limit of 0.001%.
- CaO, and SiO_2 values were variable.
- The Kakula samples were higher in Ca and Mg compared to the Kamoa Phase 6 material.

Bulk modal analysis on the minerals was conducted on each of the samples tested, of which the results are illustrated in Figure 13.45, while liberation data for each sample at a target grind of 80% passing 53 µm is presented in Figure 13.46.

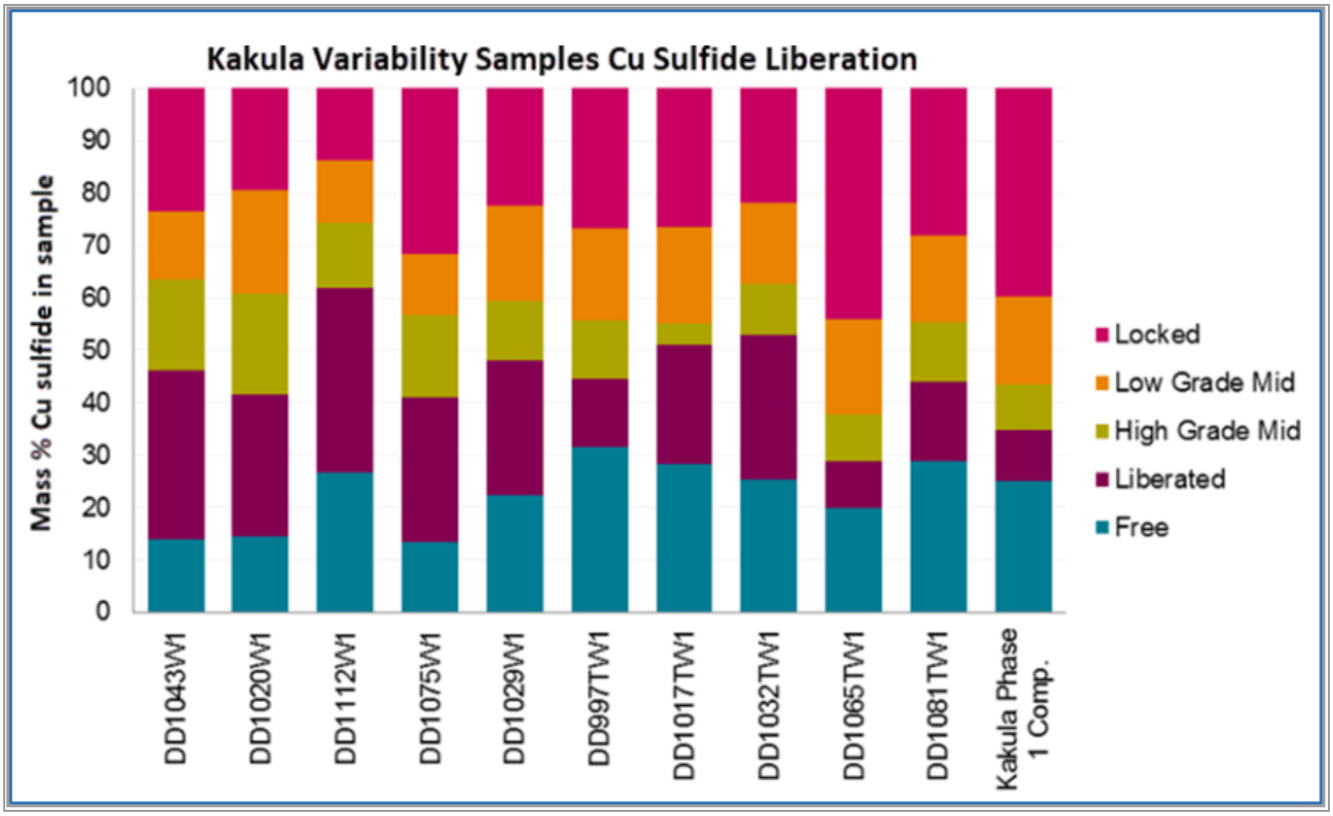
In general, the Kakula material is significantly higher in feldspar compared to Kamoa Phase 6 material. XPS reported difficulty in filtration and settling of samples, due to the fine and ultra-fine feldspar components. A varying carbonate content over the samples were noted. Chalcocite remains the main 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 sulphide Cu at 86%.

Figure 13.45 Kakula preliminary flotation variability samples mineralogy



Source: Ivanhoe 2018

Figure 13.46 Kakula preliminary flotation variability samples liberation



Source: Ivanhoe 2018

The Cu sulphide minerals occurring in the free and liberated classes, in the samples, were low at approximately 50%. This is consistent with expectations due to the fine grained nature of the sulphides. Liberation at a particle size of 80% –220 µm varied from 30–60% of the mass of Cu sulphides occurring as free or liberated grains, while the mass proportion of the locked Cu sulphides varied from 15–45%. The average Cu sulphide grain sizes varied significantly from 8–20 µm across the samples tested.

13.2.15.2 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 13.33 for a summary of the final concentrates produced in each of the tests.

Table 13.33 Kakula preliminary flotation variability results

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

The data indicated that the chalcocite rich samples produced similar results with Cu recoveries over 80% and SiO_2 grades below 10%. The sample rich in chalcopyrite (DD1065W1) only achieved an average grade of 47% Cu product at 81% Cu recovery, and high SiO_2 at 13.8%. Sample DD1075W1 was elevated in non-sulphide Cu and achieved the lowest Cu recovery at 64.7%.

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

Concentrate grades higher than 64% Cu 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.

13.2.16 Other test work

13.2.16.1 HPGR test work

In March 2018, as part of the Kakula PFS phase, ThyssenKrupp conducted HPGR (High Pressure Grinding Roll) scoping test work on Kakula material, at their testing facilities in Chloorkop, South Africa. The aim of this test work 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 test work 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 test work 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 test work alone, and pilot scale test work was required.

Following the successful scoping test work, in October 2018, ThyssenKrupp was contracted to conduct pilot plant scale HPGR test work, 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 test work 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 test work:

- 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 13.34.

Table 13.34 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

13.2.16.2 Bulk material flow test work

Bulk material flow testing was conducted by GreenTechnical, during April 2018, to facilitate 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.

13.2.16.3 Concentrate thickening test work

During July 2018, the Outotec Testing Facility in Sudbury, Canada conducted settling test work 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 characterization, flocculant selection, and batch dynamic thickening. The material characteristics, as determined by Outotec, are presented in Table 13.35.

Table 13.35 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%.

13.2.16.4 Concentrate filtration test work

Following the thickening test work, Outotec conducted further test work 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 13.36.

Table 13.36 Kakula PFS final concentrate filtration testing results summary

Dewatering technology	Test	Air drying Time (Minutes)	Filtration rate (kg DS/m2h)	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 test work indicated that the Kakula PFS final concentrate product could be successfully dewatered to within the targeted moistures (8–10%), at high solid flux rates.

13.2.16.5 Tailings thickening, rheology and filtration test work

During June 2018, SGS Canada conducted solid-liquid separation, rheology, and pressure filtration test work 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 characterization, flocculant selection, static settling, and batch dynamic thickening. The material characteristics as determined by SGS are presented in Table 13.37 and compared to the Kamoa Phase 6 tailings sample.

Table 13.37 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 13.38 for a summary of the preliminary static settling test results.

Table 13.38 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)	m2/(t/d)	58.5	58.0
Thickener unit area	m2/(t/d)	0.20	0.11
Initial settling rate	m3/m2/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 13.39 for a summary of the effect of reducing flocculant dosage rates at a fixed unit area of 0.22 m^2/t/d.

Table 13.39 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

Table 13.40 is 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 13.40 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).

13.2.17 Additional test work on Kakula Resource

13.2.17.1 Mini-Pilot plant campaign

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

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

The above mini-pilot plant test work further demonstrated the performance of the flow sheet developed during the pre-feasibility study. Open circuit batch tests, locked cycle testing, and mineralogical analyses

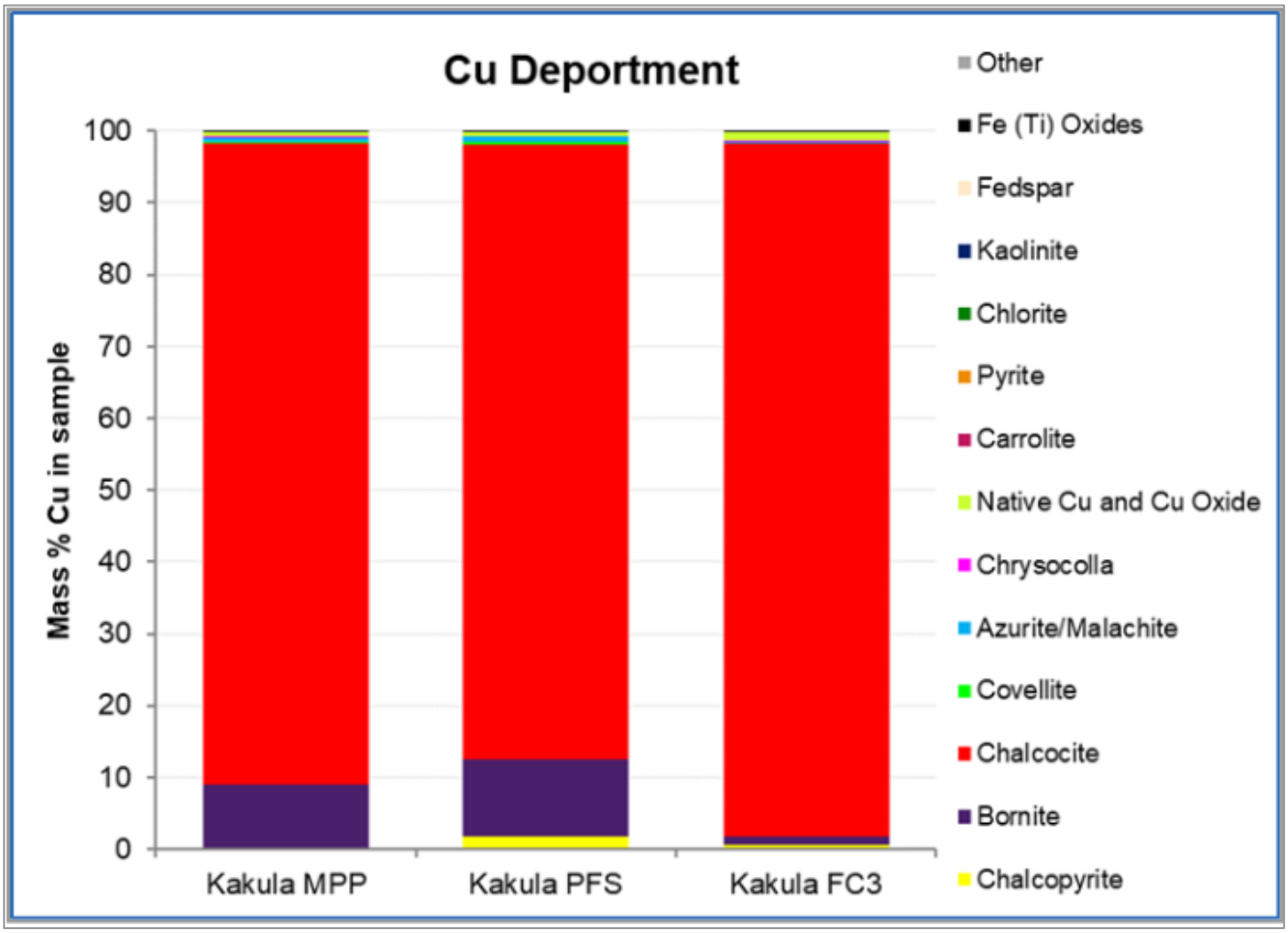
were completed to match the metallurgical performance of the mini pilot plant sample to the pre-feasibility results, prior to the pilot plant campaign.

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

13.2.17.2 Mineralogical assessment

Feed mineralogy was conducted at P80 212 µm for comparison against the PFS Kakula samples (Figure 13.47).

Figure 13.47 MPP sample mineralogy compared to previous Kakula samples



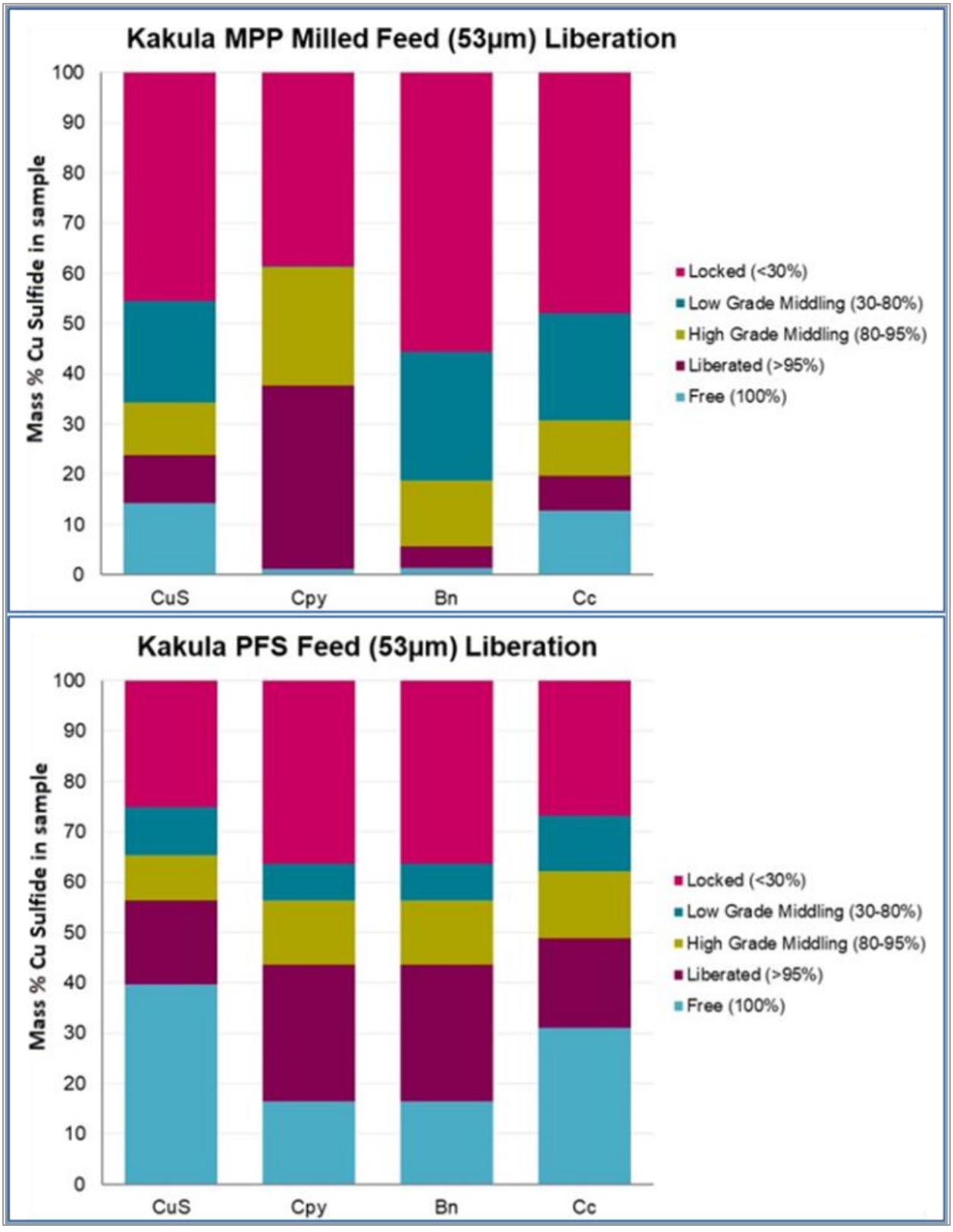
Source: Ivanhoe 2018

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 sulphide 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 sulphides.

A milled feed sample at 53 µm was analyzed (unsized) to evaluate the liberation at the target rougher feed grind. The results are summarized in Figure 13.48 below and compared against the PFS sample.

Figure 13.48 Cu sulphide liberation at 53 µm grind



Source: Ivanhoe 2018

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 sulphides was measured at 45% compared to a PFS sample of 25%.

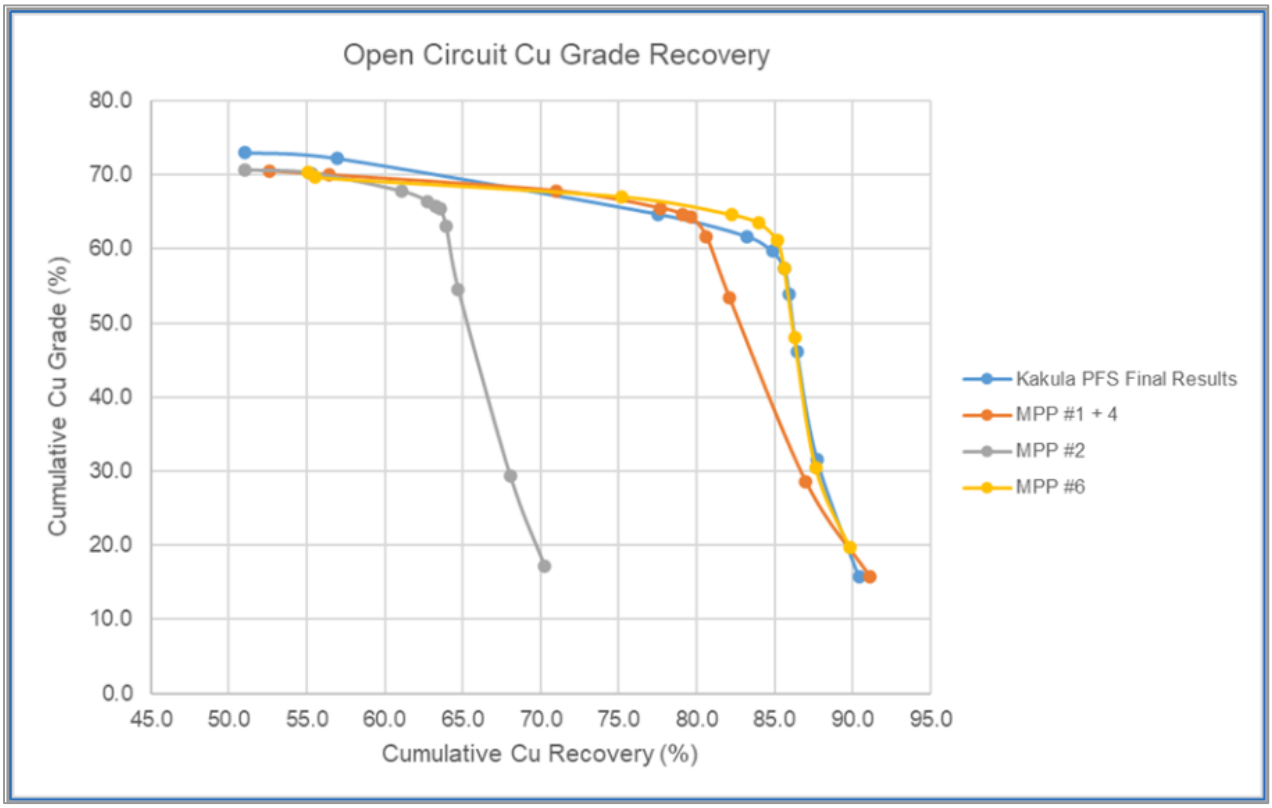
13.2.17.3 Open circuit cleaner test work

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 13.49). 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 dosage was applied to increase Cu recovery. The increase in collector dosage was motivated by the higher sample feed grade. The increase 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 13.49).

Figure 13.49 MPP cleaner circuit testing compared to PFS results

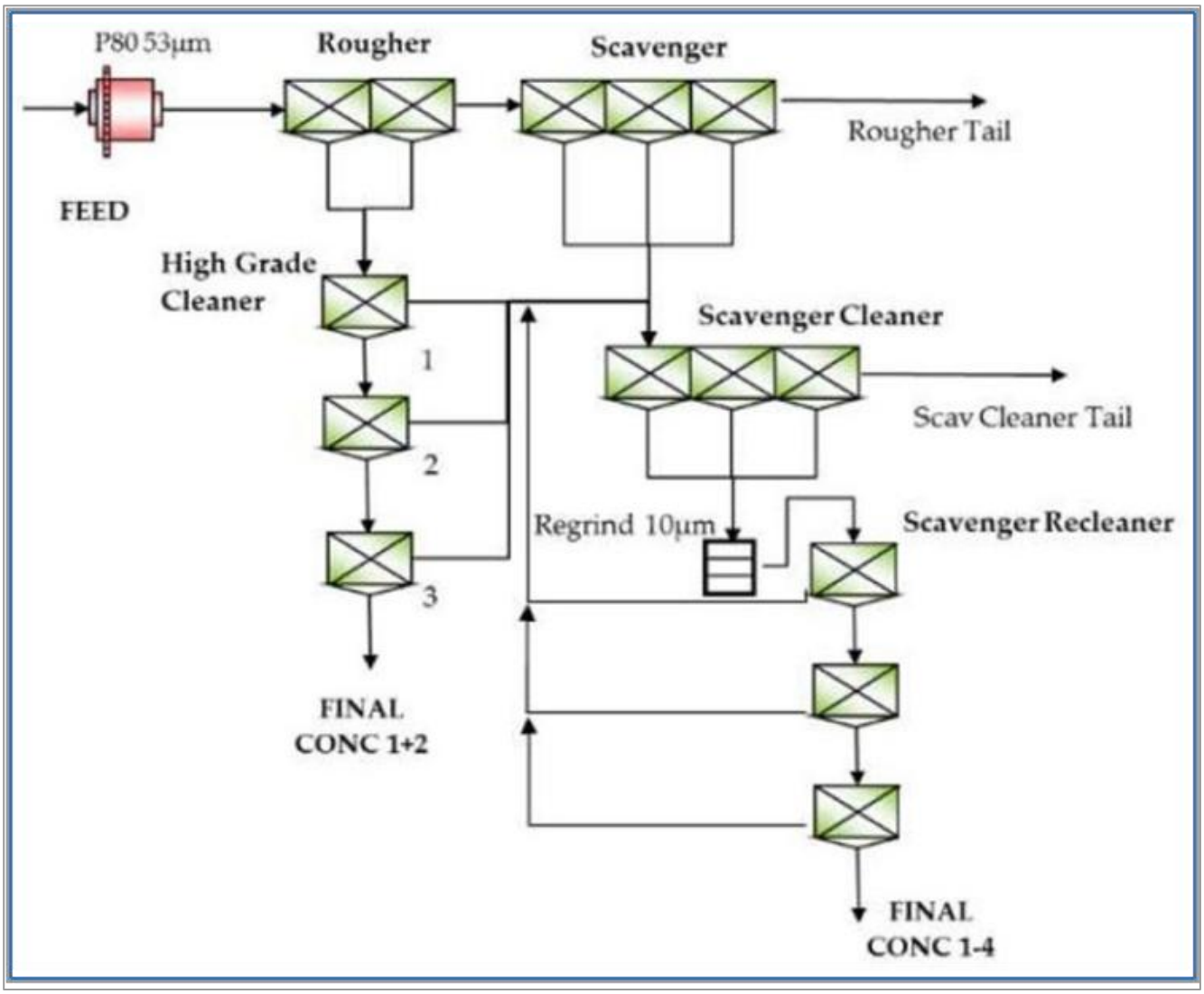


Source: Ivanhoe 2018

13.2.17.4 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 13.50).

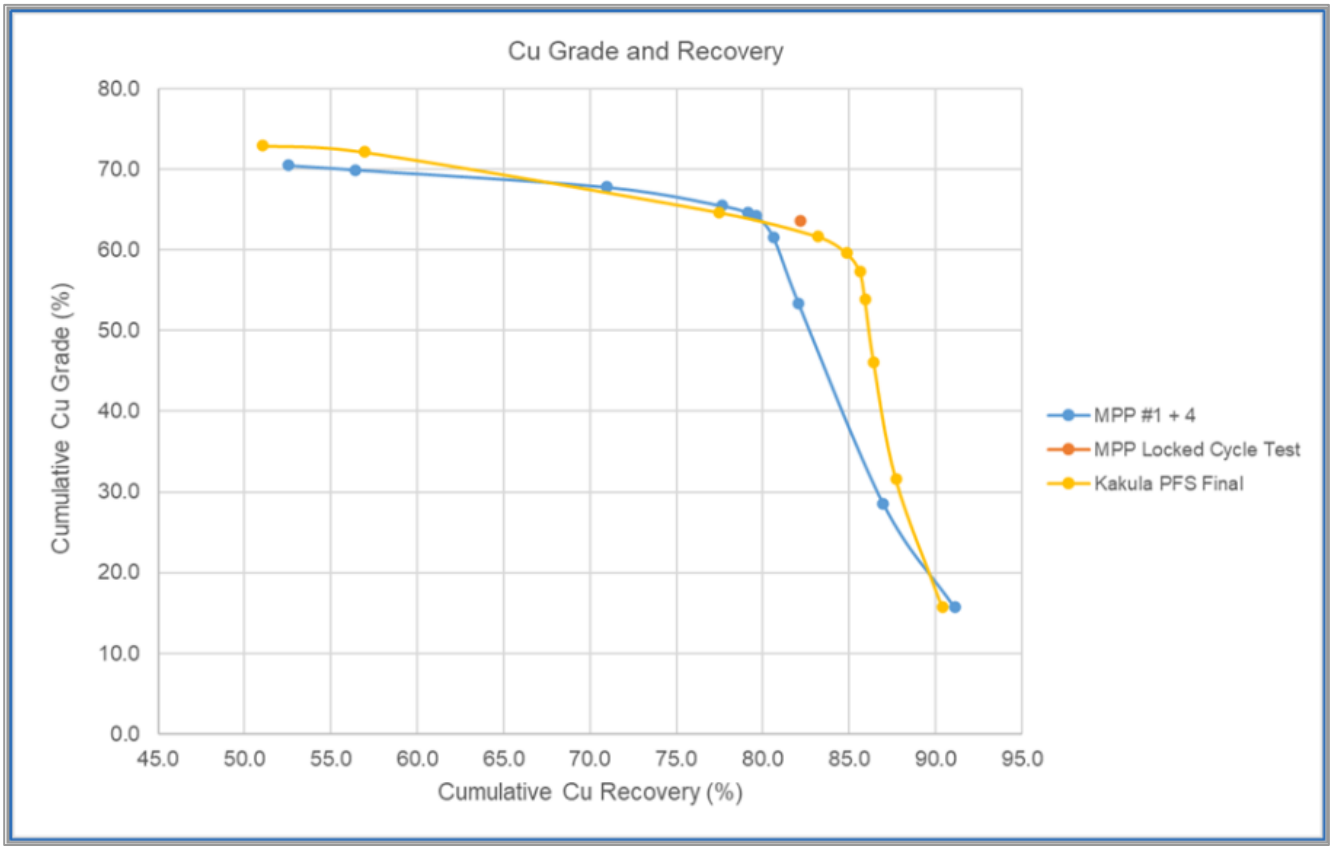
Figure 13.50 MPP locked cycle test flow sheet



Source: Ivanhoe 2018

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 13.51.

Figure 13.51 MPP locked cycle result compared to open circuit testing

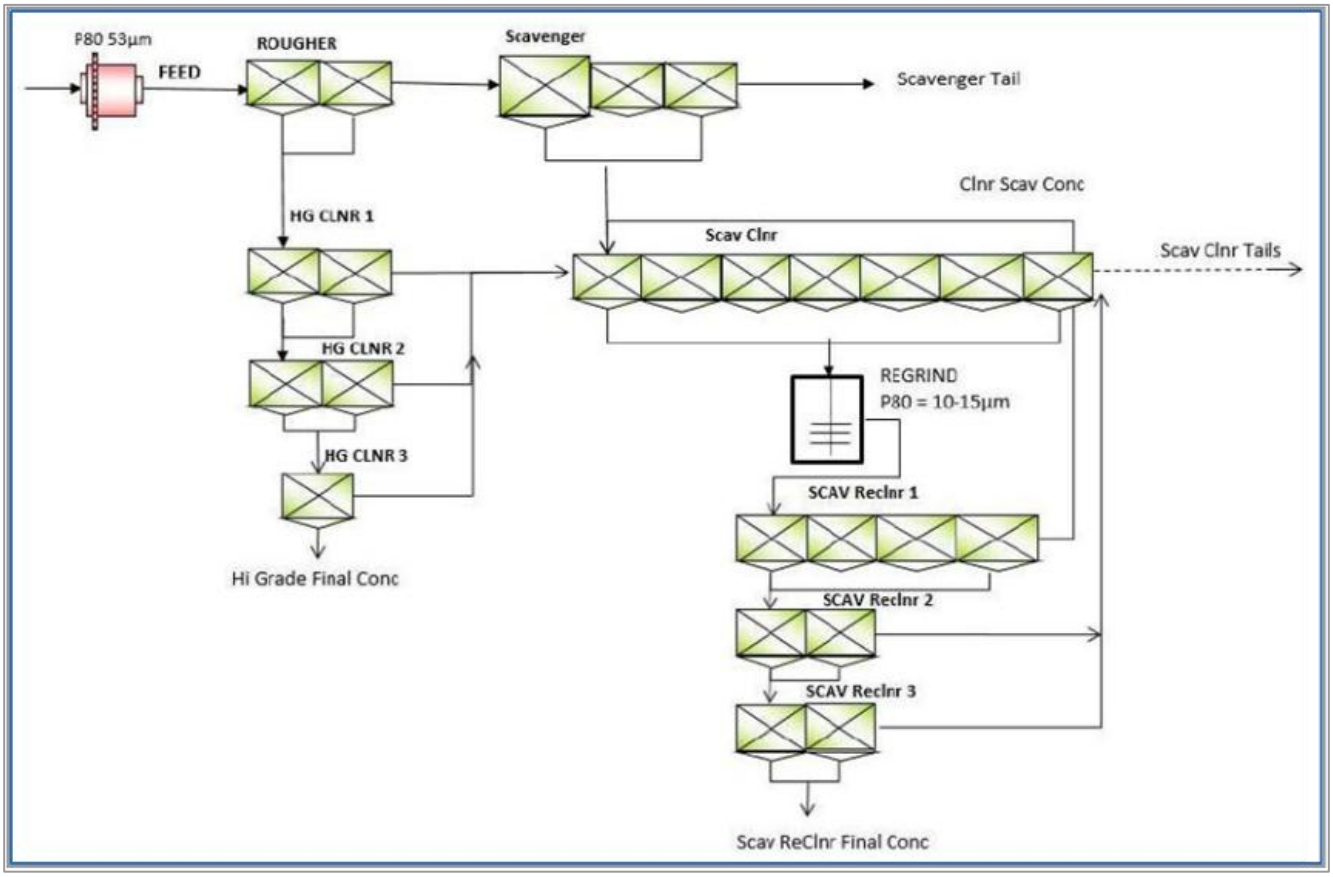


Source: Ivanhoe 2018

13.2.17.5 Backfill tailings sample generation

The first MPP run was aimed at producing a combined tailings product for backfill test work utilizing the flow sheet in Figure 13.52.

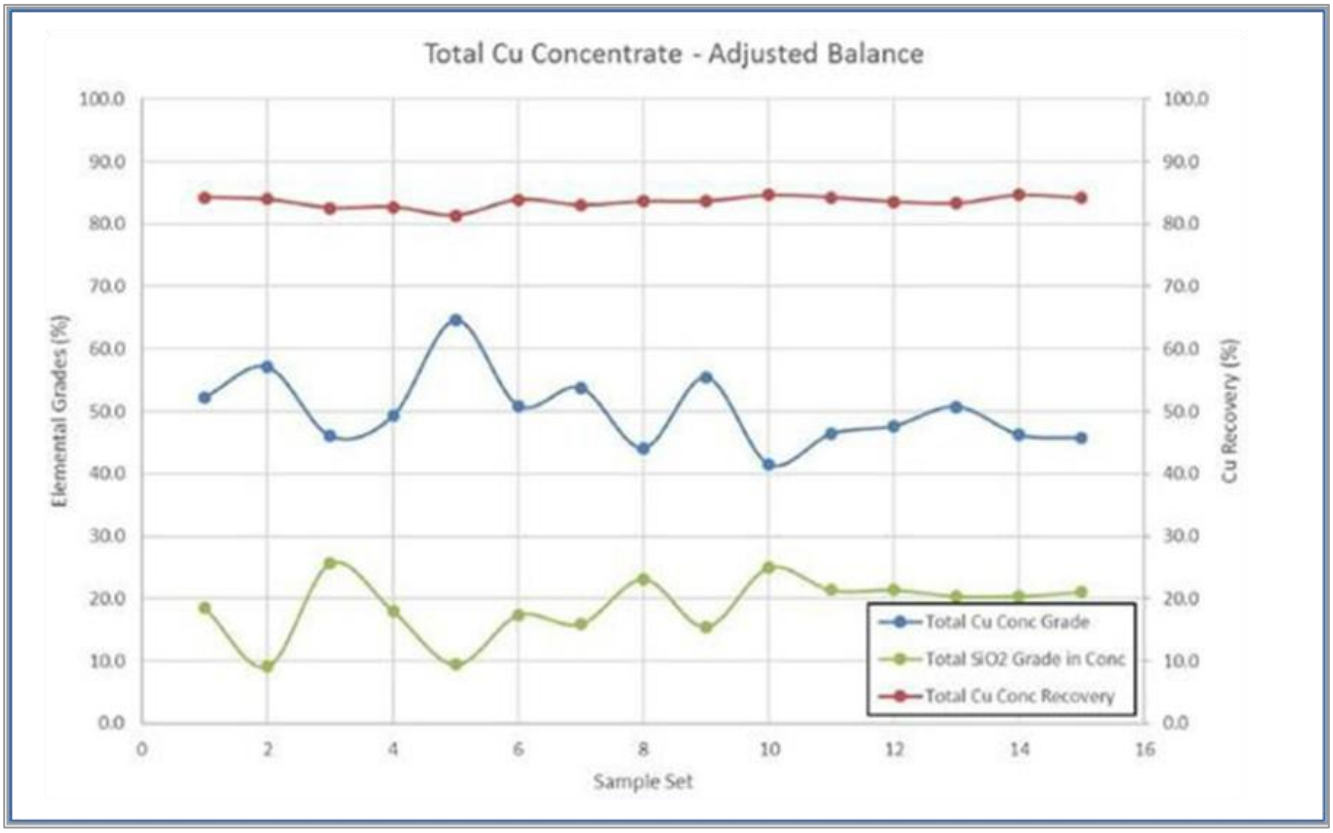
Figure 13.52 MPP run #1 flow sheet to produce combined tailings sample



Source: Ivanhoe 2018

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 summarized in Figure 13.53.

Figure 13.53 Mini-Pilot plant run #1 performance



Source: XPS, 2019.

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

13.2.17.6 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.

13.2.17.7 High-grade cleaner Jameson cell test work

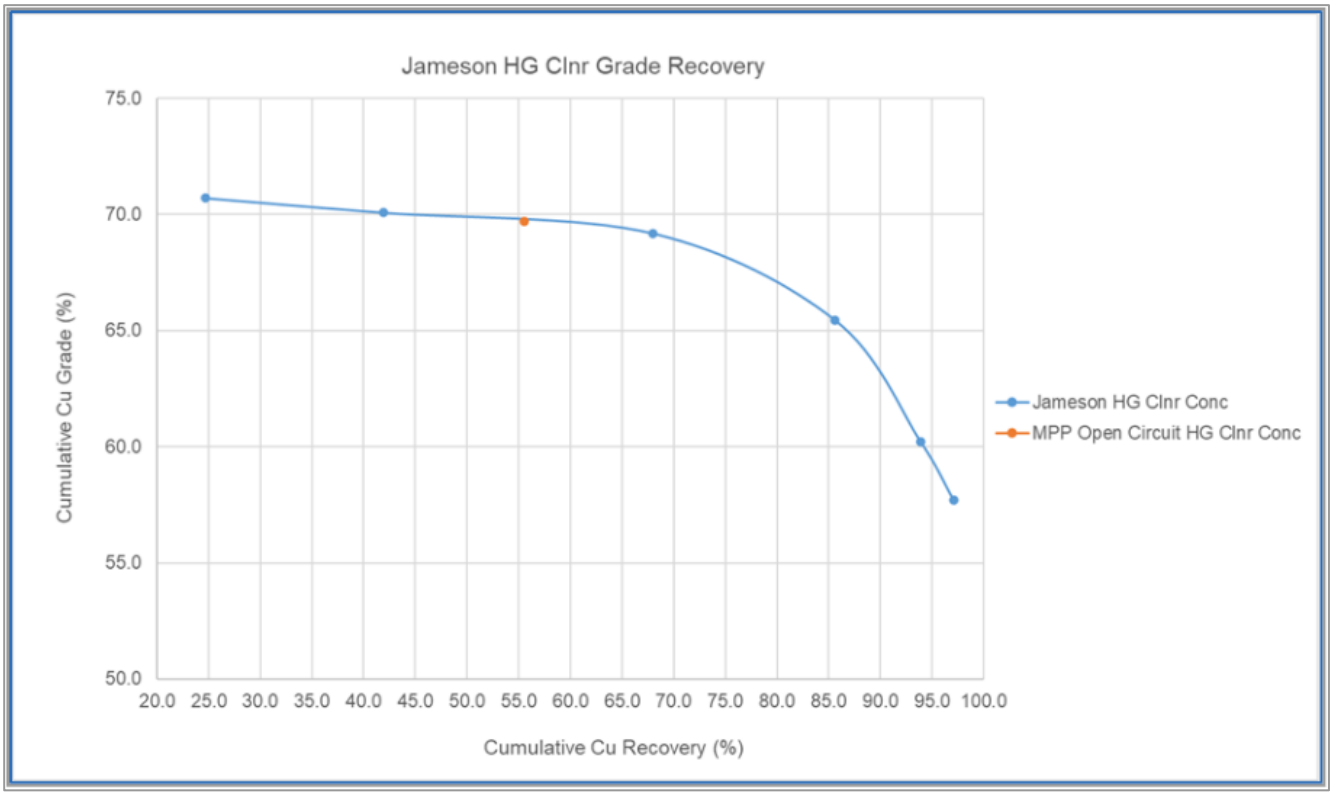
A final MPP run was conducted to produce high-grade rougher concentrate sample for Jameson cell testing, utilizing a truncated flow sheet of the mainstream flotation circuit only.

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 13.54).

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 13.54 Jameson high-grade cleaner Cu grade-recovery curve

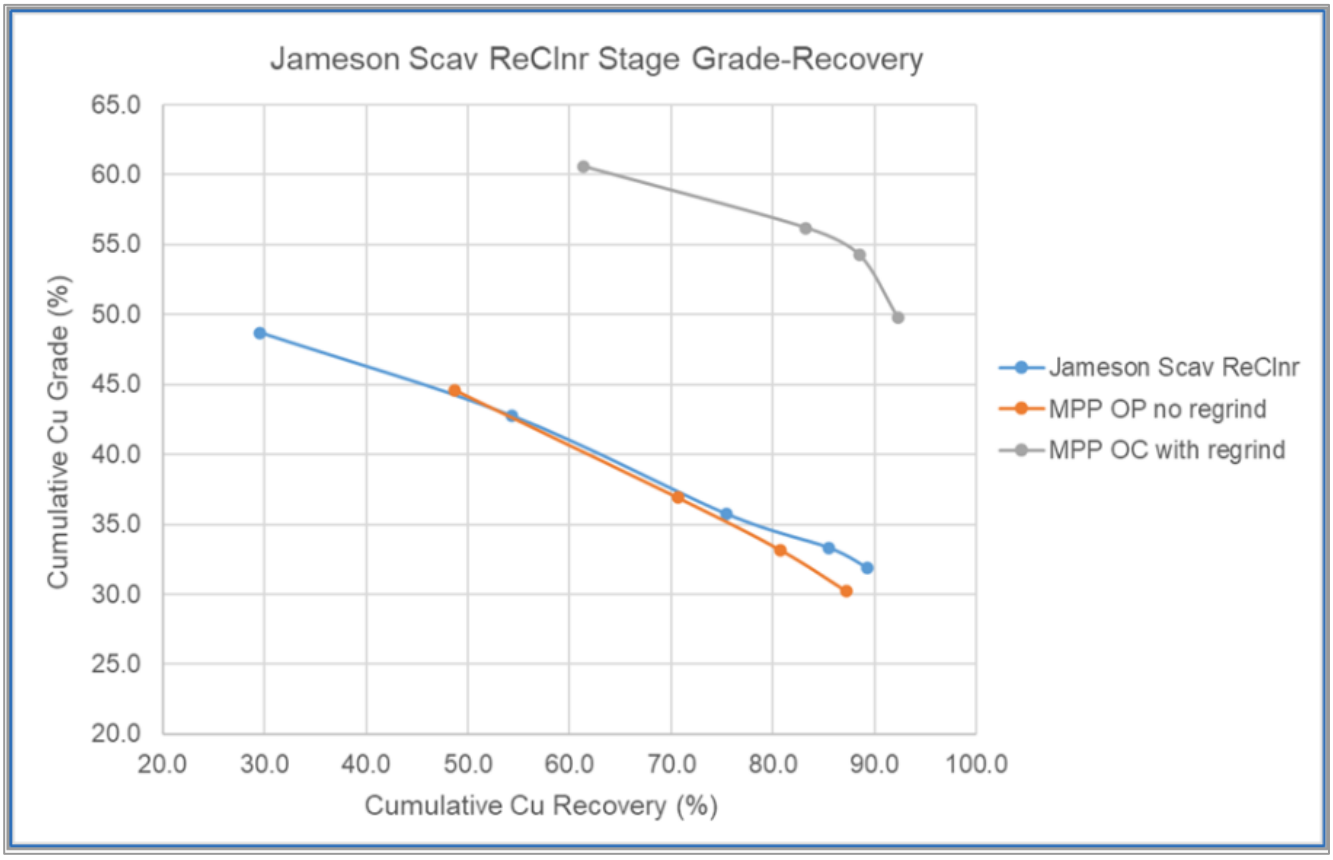


Source: XPS, 2019

13.2.17.8 Scavenger recleaner Jameson cell

A third Jameson cell test was conducted, utilizing 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 13.55).

Figure 13.55 Jameson scavenger recleaner Cu grade-recovery curve – with and without regrind



Source: XPS, 2019.

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

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

13.2.17.9 Tailings settling test work

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

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

13.2.17.10 Concentrate regrind test work

Grinding Solutions Ltd (GSL) was contracted by Metso in March 2020 to conduct hydro-cyclone and signature plot test work 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 are summarized in Table 13.41.

Table 13.41 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 P80 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 P80 10 µm. The signature plot summary is shown in Table 13.42.

Table 13.42 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

13.2.17.11 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.

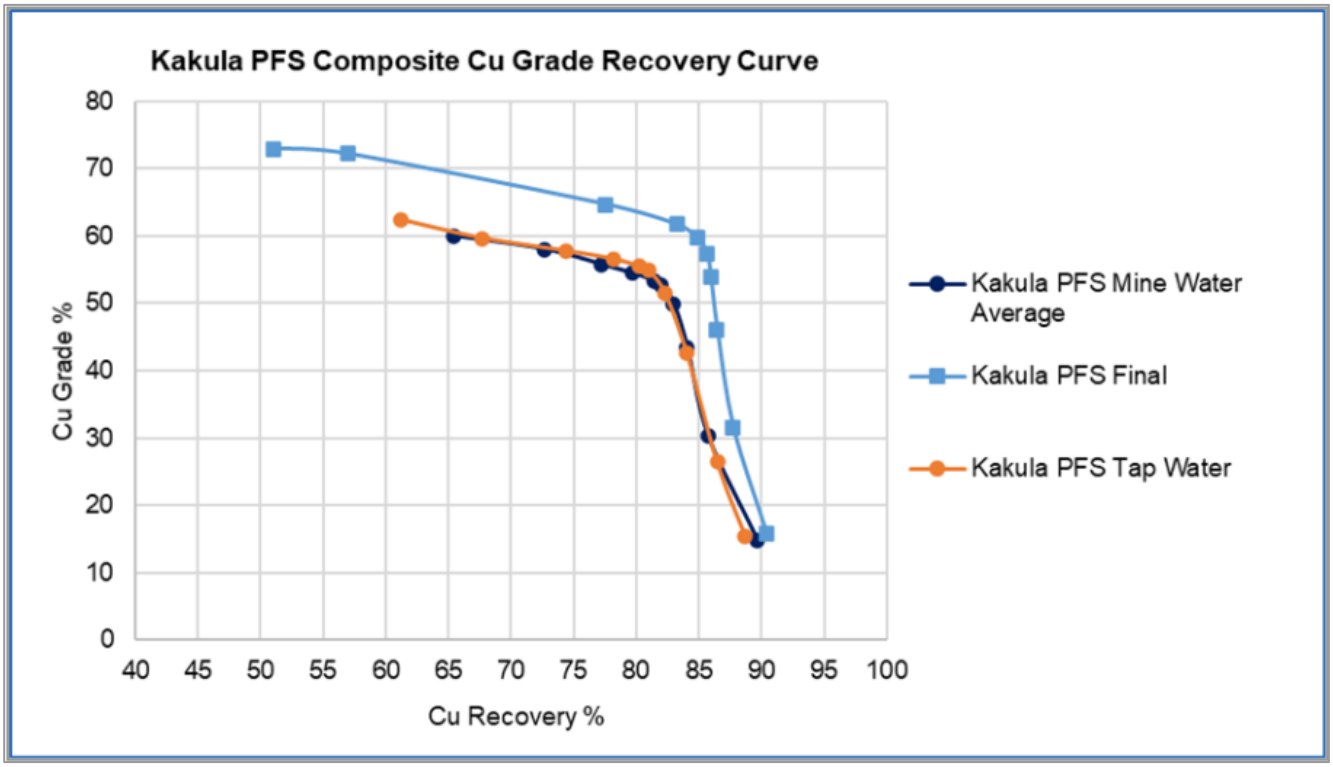
Table 13.43 Flotation results using mine water

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

Figure 13.56 Effect of mine water flotation testing on Kakula PFS composite sample



Source: XPS, 2019

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

13.2.18 Testwork on Kakula West

13.2.18.1 Preliminary test work on Kakula West material

In 2018, XPS conducted mineralogy and flotation tests on a single 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.

13.2.18.2 Kakula West sample details and characterization

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 13.44. This material was composited into a single sample for testing. Head analyses were conducted in triplicate, on the Kakula West composite sample, and is summarized in Table 13.45.

Table 13.44 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 13.45 Kakula West flotation composite sample head analysis

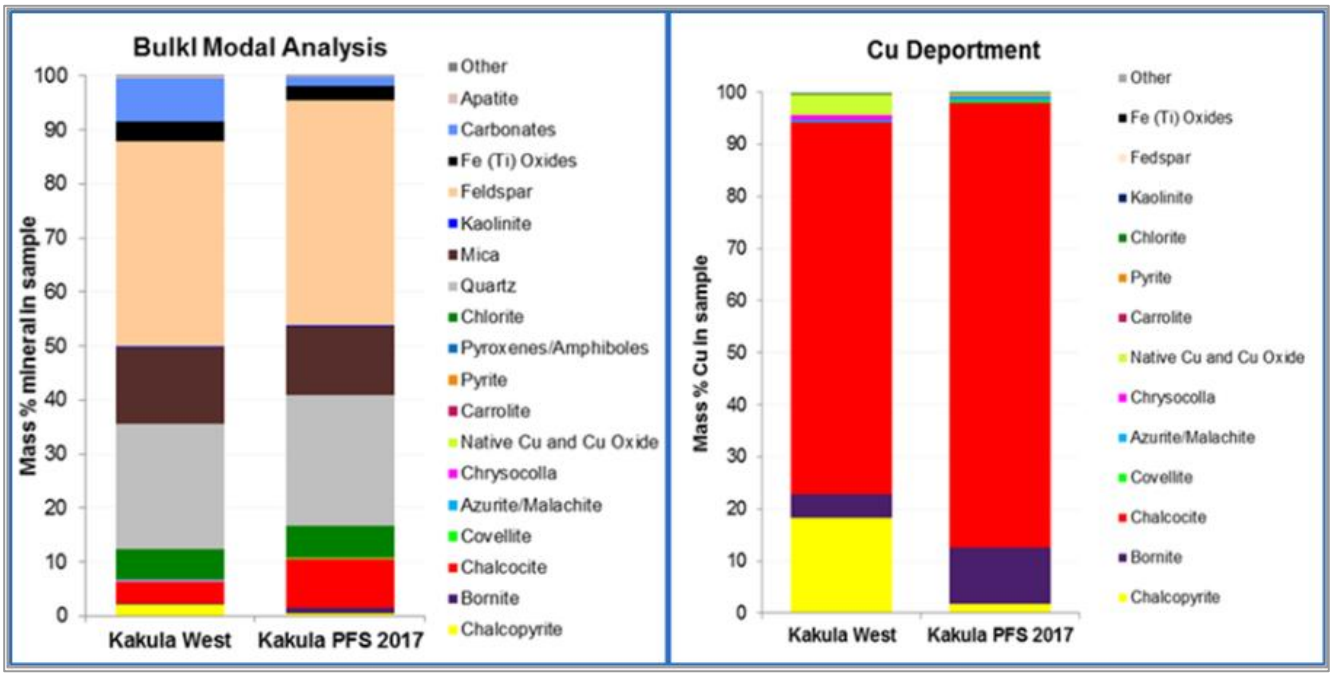
Sample	Cu (%)	S (%)	SiO_2 (%)	Fe (%)	Al_2O_3 (%)	CaO (%)	MgO (%)	As (%)
Kakula West Flotation composite sample	3.17	1.07	54.00	4.99	12.90	4.62	4.50	<0.01

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 13.57.

The Kakula West sample was lower in Cu grade compared to the Kakula PFS sample tested. The main Cu mineral in the Kakula West material was chalcocite, followed by chalcopyrite and smaller amounts of bornite. The Kakula West sample hosts higher levels of chalcopyrite than the Kakula PFS sample tested.

The Kakula West and Kakula PFS samples had similar levels of chlorites, quartz, and mica. The Kakula West sample showed slightly lower feldspar levels when compared to the Kakula PFS sample, but with higher carbonates. The average grain size of the Kakula West Cu sulphide minerals was noted as similar to the Kamoa Phase 6 sample – slightly coarser than the Kakula PFS sample tested.

Figure 13.57 Kakula west sample mineralogy



Source: XPS 2019

13.2.19 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.

13.2.20 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 test work, however indicated that the Kakula and Kamoa material can be treated in a common concentrator.

13.2.21 Kamoa-Kakula 2025 PFS recovery estimate

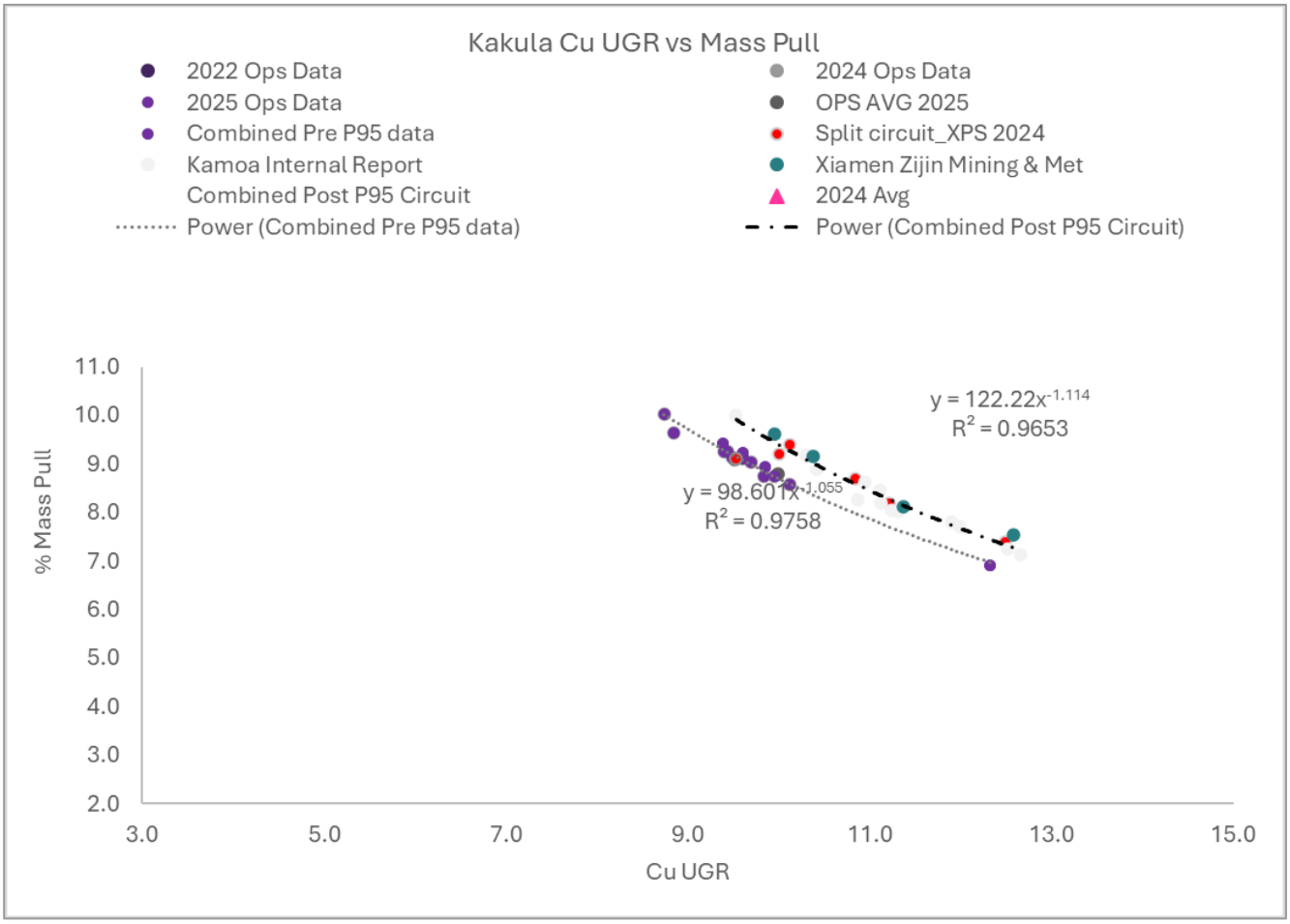
13.2.21.1 Kakula

The Kamoa-Kakula 2025 recovery estimate for Kakula material is based on the test information generated by the Kakula Phase 1 campaign, Kakula variability test work, current steady state operating data and Kakula recovery optimization test work results from XPS and ZIJIN laboratories.

The Kakula recovery model targets a final product grade of 47.0% 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 Figure 49). Targeted UGRs was calculated by dividing the targeted final product grade (47.0%) by the individual back calculated head grades from each of the tests, and the associated mass pulls noted.

Current, monthly operating data for the Phase 1 and Phase 2 concentrator modules was also plotted on the same graph with the modelled data to validate the model. The Kakula West test work datapoint was further plotted to assess applicability. The data is presented in Figure 13.58.

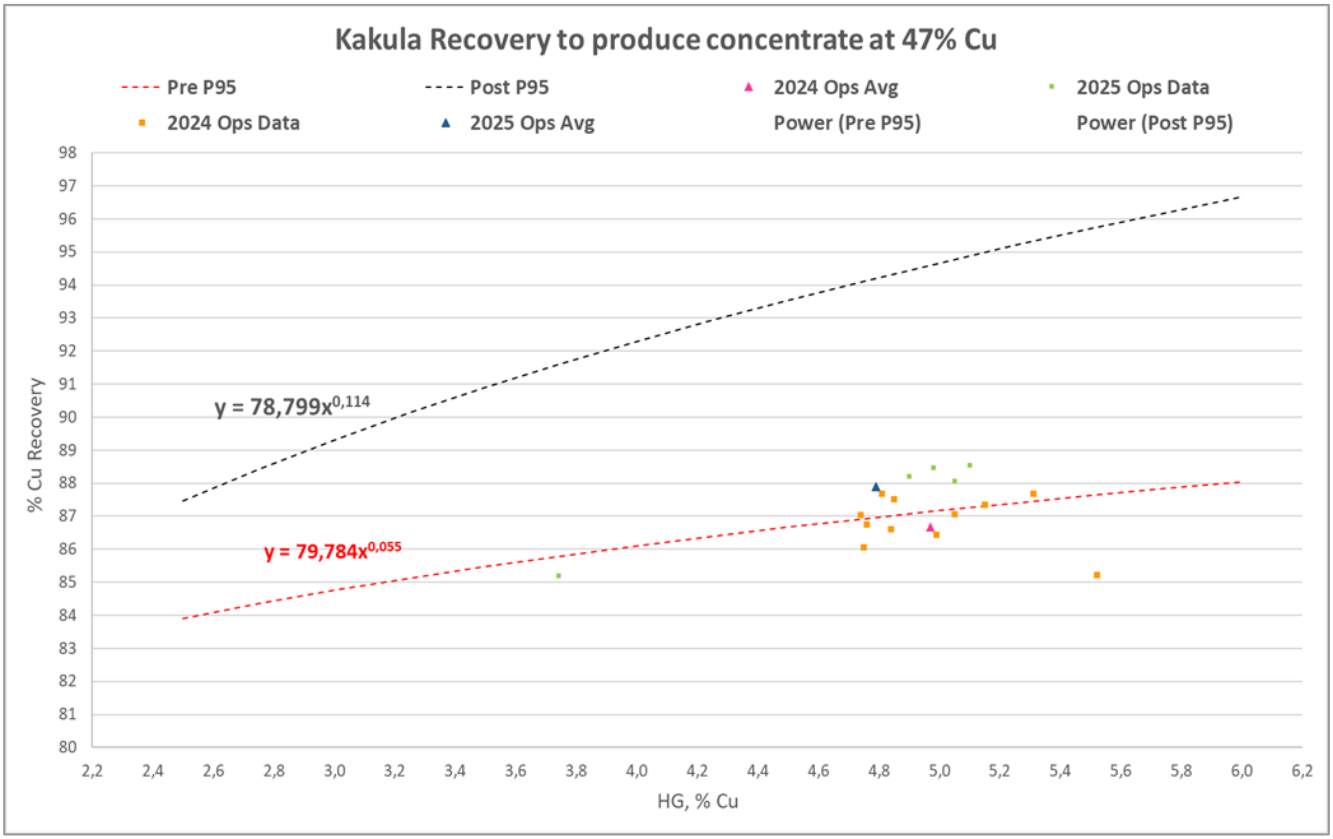
Figure 13.58 Mass pull as a function of Cu upgrade ratio for 47% Cu concentrate



Source: XPS 2019
Note: 1 y= mass pull (%) and x=Upgrade Ratio

The resulting correlation from Figure 13.59 was used to calculate the expected mass pull for varying head grades, by determining the targeted Cu upgrade ratio based on a 47.0% final product. The associated Cu recovery is then calculated using the mass pull and concentrate grade. The Kakula Cu recovery algorithms for the baseline flows sheet and the split flotation circuit (P95) are shown in Figure 13.59. The split flotation circuit shows improvement in performance.

Figure 13.59 Kakula Cu recovery as a function of Cu head grade



Source: XPS 2026
Notes: [1] y=Cu recovery % and x= Cu head grade

The head grade and recovery correlations were applied to the Kakula-Kamoa 2025 PFS life-of-mine production plan.

- For the current Kakula operation Recovery = 79.784x(Cu grade)^-0,55
- Post optimization project (P95) commissioning Recovery = 78.799x(Cu grade)^0.114

13.2.22 Kamoa

The Kamoa Phase 3 variability test work results were used to generate the upgrade ratio to mass pull relationships from which the recovery models (recovery vs head grade) for both supergene and hypogene ores were developed. The hypogene samples selected are listed in Figure 13.59.

Table 13.46 Kamoa hypogene variability test samples

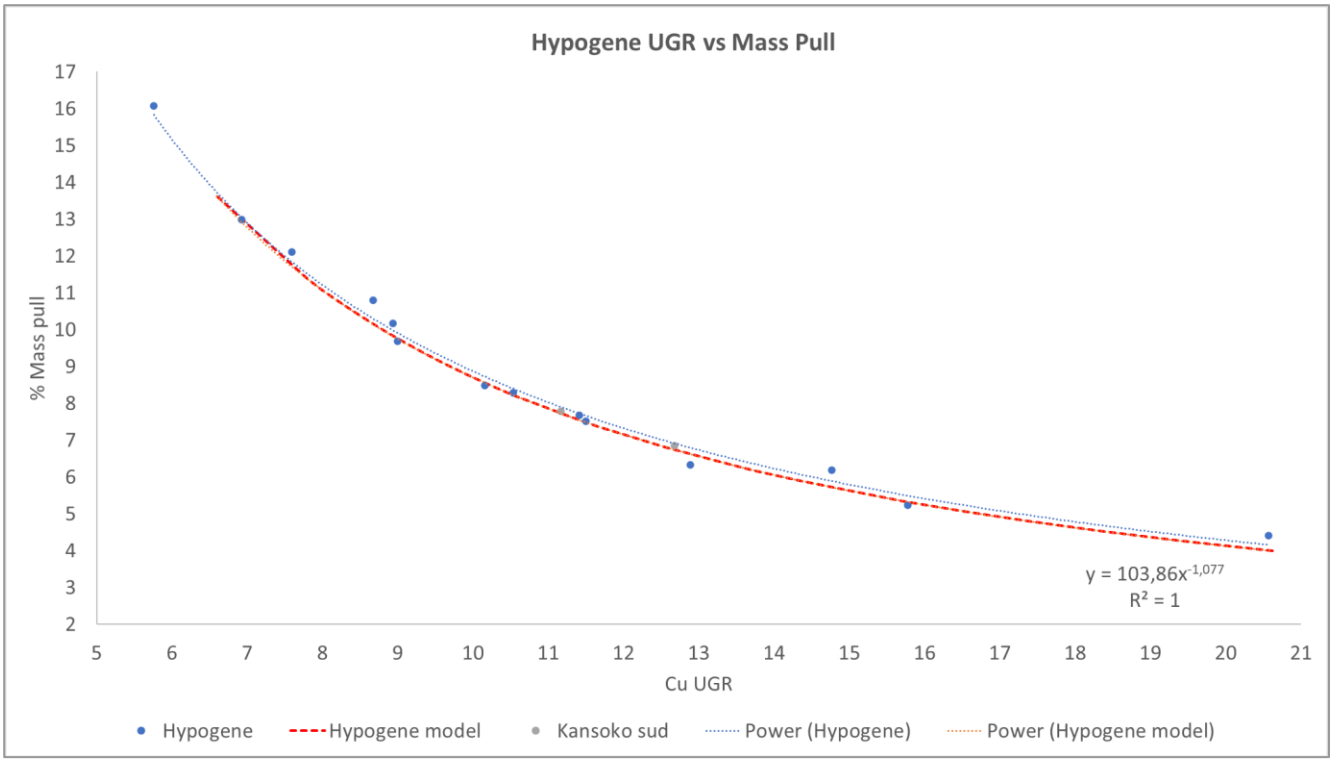
Description	Deposit
IFS4a 6A Hypo Average	Kansoko sud
K1_DD1725 T1	Kamoa 1 Variability
K1_DD1725 T2	Kamoa 1 Variability
K1_DD1727 T1	Kamoa 1 Variability
K1_DD1727 T2	Kamoa 1 Variability
K1_DD1728 T1	Kamoa 1 Variability

Description	Deposit
K1_DD1728 T2	Kamoa 1 Variability
K1 composite	Kamoa 1 Master Comp
K2_DD1710 T1	Kamoa 2 Variability
K2_DD1710 T2	Kamoa 2 Variability
K2_DD1712 T1	Kamoa 2 Variability
K2_DD1712 T2	Kamoa 2 Variability
K2_DD1716 T1	Kamoa 2 Variability
K2_DD1716 T2	Kamoa 2 Variability

Figure 13.60 shows the upgrade ratio to mass pull relationship where:

$$Upgrade\ Ratio\ (y) = 103.86 \times Mass\ Pull\ (x)^{-1.077}$$

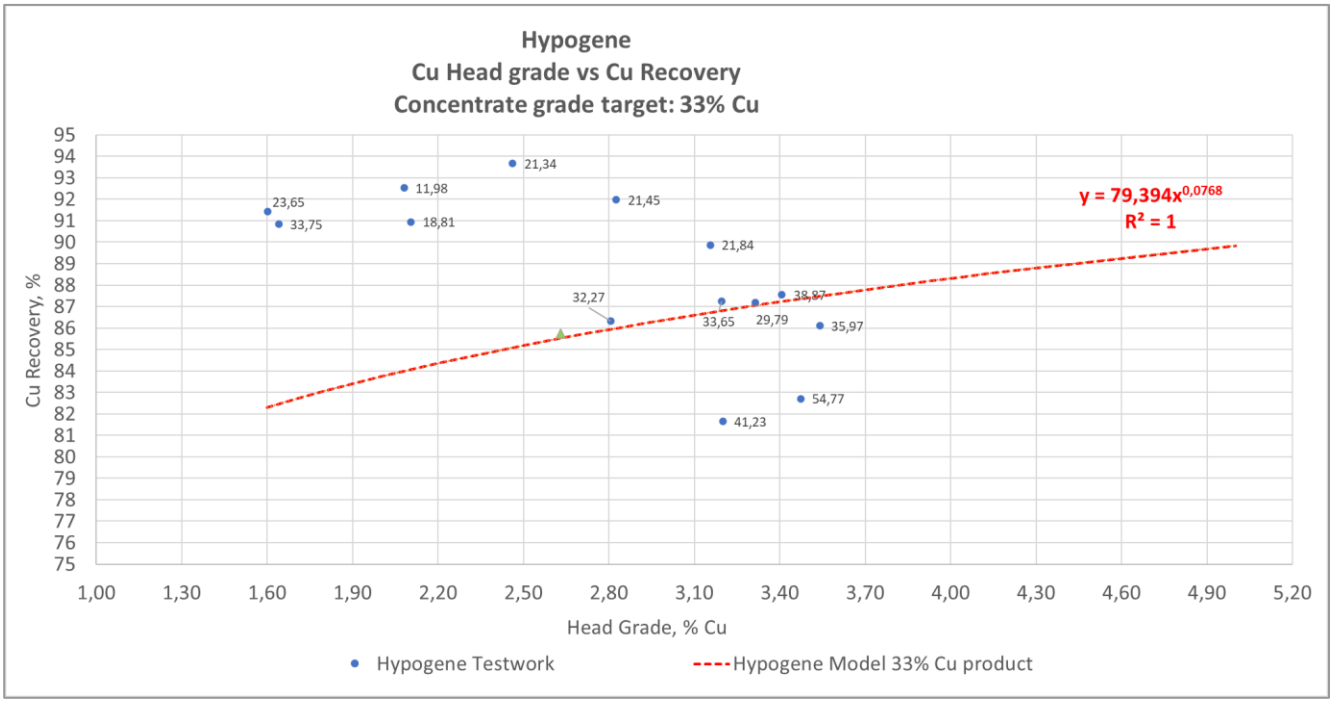
Figure 13.60 Hypogene upgrade ratio to mass pull curve



Source: XPS 2019

Figure 13.61 shows the recovery head grade relationship, targeting a concentrate grade of 33% Cu.

Figure 13.61 Kansoko, Kansoko Sud Kamoa 1-6 Hypogene Recovery as a function of head grade



Source: DRA 2026

The samples used for the supergene recovery model are listed in Table 13.47.

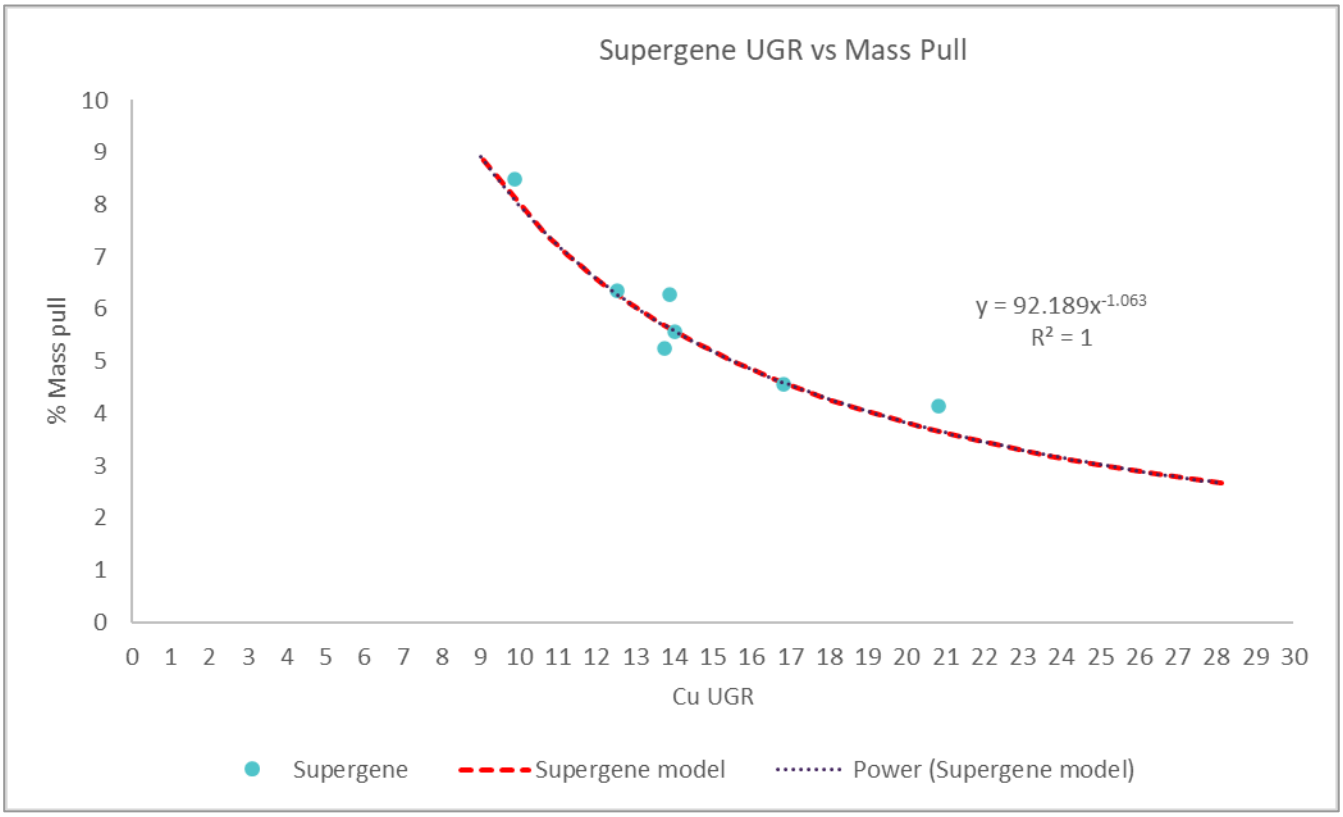
Table 13.47 Kamoa supergene variability samples

Description	Deposit
IFS4a 6A Super Average	Kansoko sud
K2_DD1708 T1	Kamoa 2 Variability
K2_DD1708 T2	Kamoa 2 Variability
K2_DD1720 T1	Kamoa 2 Variability
K2_DD1720 T2	Kamoa 2 Variability
K2_DD1724 T1	Kamoa 2 Variability
K2_DD1724 T2	Kamoa 2 Variability

Figure 13.62 shows the upgrade ratio mass pull relationship where:

$$Upgrade\ Ratio\ (y) = 92.189 \times Mass\ Pull\ (x)^{-1.063}$$

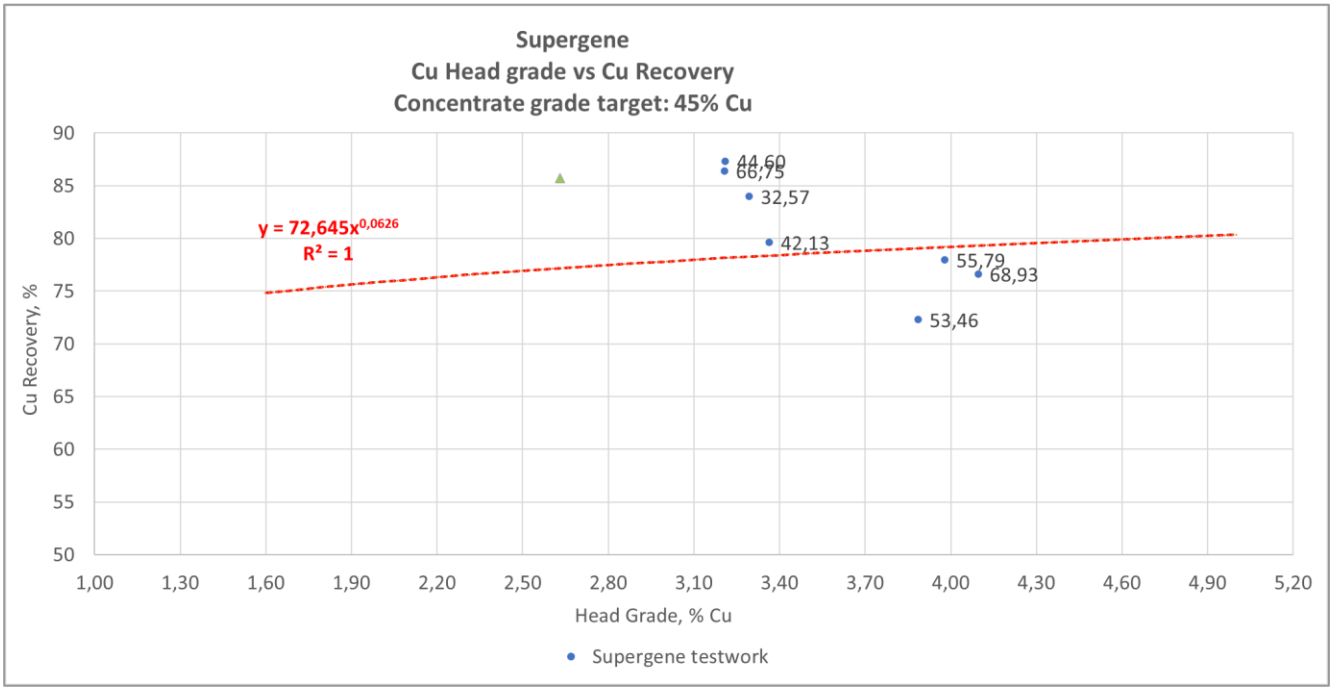
Figure 13.62 Supergene upgrade ratio to mass pull curve



Source: XPS 2019

Figure 13.62 shows the recovery head grade relationship, targeting a concentrate grade of 45% Cu.

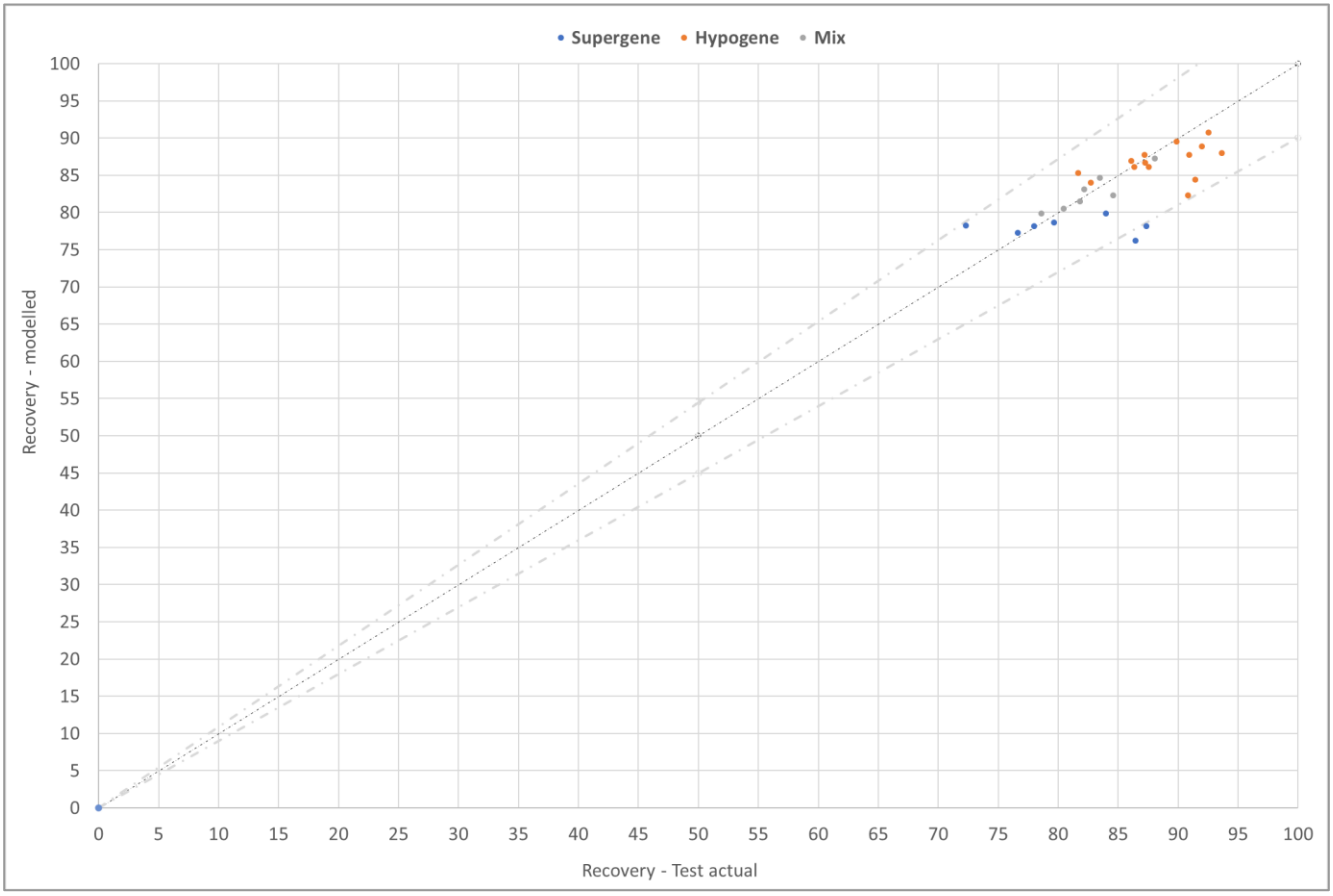
Figure 13.63 Kamoa supergene Cu recovery as a function of head grade



Source: DRA 2026

The recovery model was compared to the actual test results. This shows a close correlation, with the actual test results showing clusters around the model curve. This is illustrated in Figure 13.64.

Figure 13.64 Recovery model compared to test results



Source: XPS 2019

The Kamoa recovery model results are summarized in Table 13.48.

Table 13.48 Kamoa recovery model results

Supergene	
Concentrate Grade % Cu	45
Recovery Formula	Recovery = 72.645 x TCu feed grade^$^{0.0626}$
Hypogene	
Concentrate Grade % Cu	33
Recovery Formula	Recovery = 79.394 x TCu feed grade^$^{-0.0768}$

13.3 Flash smelting pilot plant test work

The flash smelting pilot test work addressed the following:

- The copper concentrates from Kakula and Kamoa have quite different mineralogy.
- The Kakula concentrate is chalcocite rich, high in copper, low in sulphur and iron and energy deficient.

- The Kamoa concentrate is chalcopyrite and bornite rich, low in copper, high in sulphur and iron and high energy.
- In the flash smelting environment adjustments must be made to accommodate the variances that can result, these include:
  - Slag production due to the amount of iron that must be oxidized and fluxed together with the concentrate gangue contents.
  - Flux additions, mainly lime is dependent on iron and gangue contents.
- Fuel consumption is high with low energy concentrates and low with high energy concentrates. Energy in the concentrates is derived from oxidation of copper, sulphur and iron.
- The propensity of the slag to foam depends on the oxidation potential maintained in the DBF. If the copper in the slag is too low, there is a tendency for sulphur to remain in the blister and to suddenly oxidize releasing sulphur dioxide gas under the slag and causing it to foam. Higher copper in slag inhibits this effect due to the oxidation potential being higher.

The test work covered the following range of concentrate composition:

- From Kamoa: Kakula 79:21
- To Kamoa: Kakula 35:65

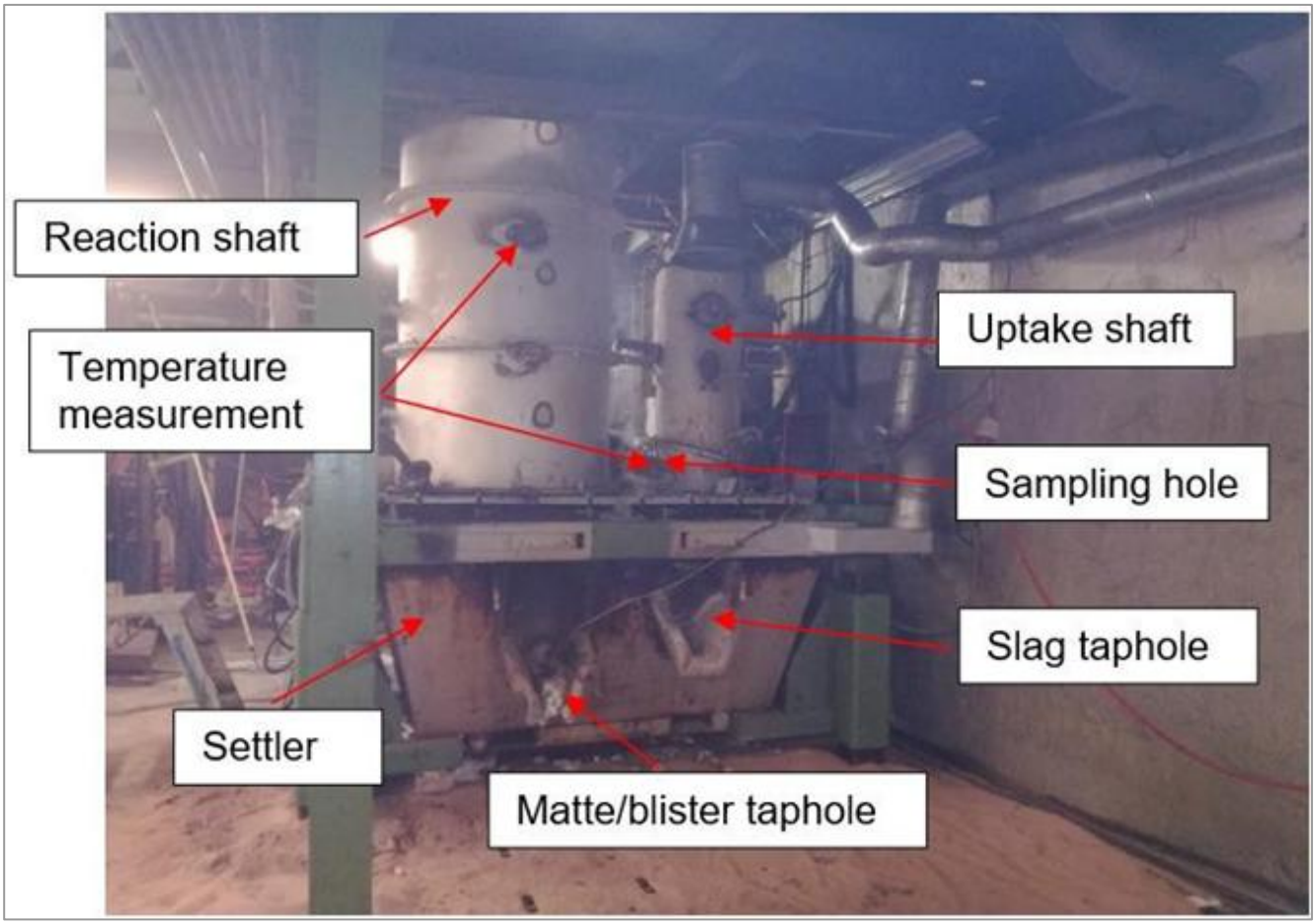
Since Kamoa concentrate was not available at the time the test work was conducted a proxy chalcopyrite rich concentrate was used instead.

At the upper end, the test work covered the maximum projected ratio of Kamoa to Kakula concentrate 70:30. At the lower end it was short of the minimum projected ratio of 20:80 but it is understood that with appropriate fuel additions such low energy concentrates can be smelted

13.3.1 Pilot plant description

The pilot plant unit used is shown below in Figure 13.65.

Figure 13.65 Pilot Plant



Source: Metso

The reaction shaft had a diameter of 400 mm and the uptake shaft 200 mm and the unit was capable of an average concentrate feed rate of 140 kg/hr. 8 tests were conducted at various Kamoa: Kakula ratios as noted above. Lime additions were also varied to optimize the slag chemistry.

13.3.2 Pilot plant campaign outcomes

Key findings from the test work were as follows:

- Fe / SiO_2 ratio in the DBF slag should ideally be above 0.5. Slag viscosity and propensity to foam are very dependent on this ratio.
- With low proportions of Kamoa concentrate it may be difficult to achieve Fe / SiO_2 ratios above 0.5. This needs to be compensated by increasing the CaO / SiO_2 ratio and allowing higher copper content in the DBF slag. Alternatively, pyrite can be added with the added benefits of providing an energy source and producing additional sulphuric acid.
- Sulphur content in the DBF blister needs to be kept below 0.2% to avoid foaming. This is achieved by keeping copper in slag in the range 16 - 20 %. In turn this needs to be controlled by having the correct Oxygen coefficient (Nm^3 / ton concentrate).
- Blister temperature typically 1200 – 1250°C. Slag temperature typically 1300 – 1350°C.

Some useful operating correlations were presented relating slag viscosity and propensity to foam against DBF slag composition.

Distribution coefficients for deleterious elements between slag and blister were determined. It is expected that anode quality will be very good because the deleterious elements in the feed are very low

For the most part the DBF technology is well suited to the concentrates produced within the projected range of Kamoa / Kakula ratios.

Once the Kamoa / Kakula ratio exceeds 70:30 there may be a requirement to reduce throughput or modify the smelter.

13.4 Comments on section 13

In the opinion of the QP, the metallurgical test work conducted for the Kakula and Kamoa deposits is sufficient for PFS and feasibility level process design respectively. The samples tested are representative of the target mining area. The comminution characteristics are well established and have consistency across the various testing phases and across the prospective mining areas. Sufficient work has been conducted to design the regrind circuits.

The mineralogy is variable, but the flotation characteristics are well understood and explainable in terms of process mineralogy. The samples tested reasonably represent the material to be mined and processed according to the mine schedule.

The project mineralized zones do not contain deleterious elements often found in copper concentrates, such as arsenic and fluorine. As a result, the flotation test work has consistently generated concentrates that are free of penalty elements.

The pervasive presence of ultrafine copper sulphides in all Kamoa samples leads to strong recovery of silica through attachment with these sulphides. This, in turn, has led to high rougher mass pull rates and silica rejection challenges in final concentrate production, which is mitigated to a large degree by 10 µm regrinding of middling streams. The most recent test work, 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 and Kamoa circuits further facilitated 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 Kamoa regrind power requirement has been confirmed by test work as described in Section 13.2.5.3.

The prediction of copper recovery for Kamoa hypogene samples and the Kansoko surface linked oxidation supergene samples is reasonable based on variability test work. This is an improvement to the 2023 PFS which estimated recovery on assumed unfloatatble Cu. The model match operating data.

The prediction of copper recovery for the Kakula material is based on variability test work 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 mineralized zones, the Kakula deposit has less variability in copper mineralization, 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 mineralized units to construct a full three-dimensional (3D) block model with multiple horizontal domains stacked vertically. Mineralized 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 mineralized domains were modelled within different stratigraphic horizons, including the Bonanza Zone mineralization which is hosted within the KPS. At Kakula, a single mineralized 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 pale green siltstone, basal mineralized 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 mineralized 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 mineralization 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 specific gravity (SG) data were exported from the Ivanhoe acQuire database as a series of CSV files, imported into Datamine Studio RM software (Datamine), and combined to form a de-surveyed drillhole file for each deposit area.

14.2 Selective mineralized zones

14.2.1 Kamoa

In general, the selective mineralized 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 localized 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 that host the majority of the Kamoa Mineral Resource. 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 mineralization in the Upper SMZ weakens, and a separate mineralized 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 recognized 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 makes correlations with Kakula difficult; however, the mineralization 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, mineralization 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 mineralized 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 mineralized zone was modelled in these areas.

The assay file, with the SMZ selections flagged, was then imported into Datamine 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

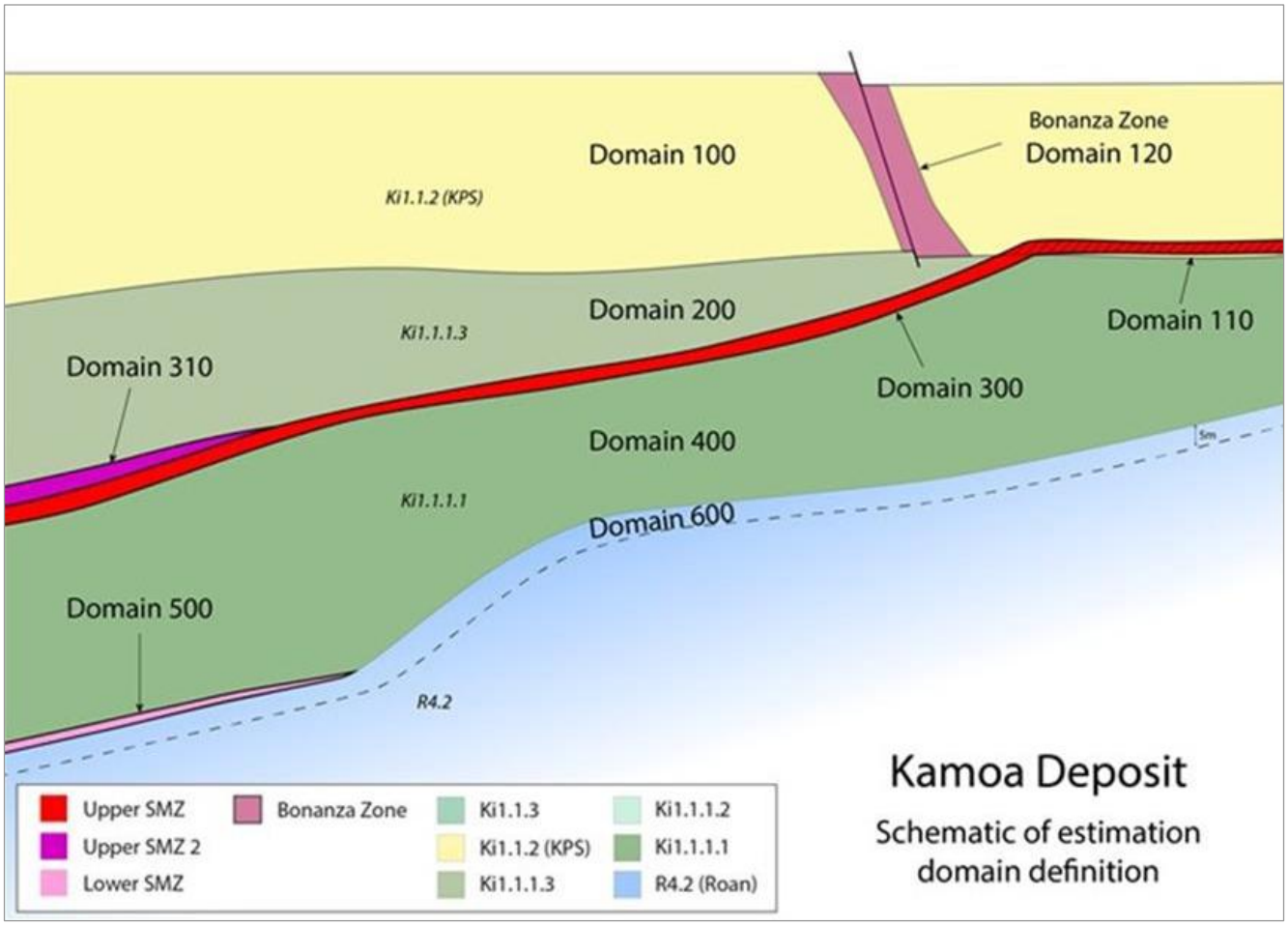
At Kakula, the highest copper grades are located just above the Roan contact in the basal siltstone. Grades usually drop sharply in the overlying diamictite and in general decrease gradually with increasing elevation. For resource estimation, the mineralized zone was defined using an approximate 1% TCu cut-off. No minimum thickness criteria were applied during coding of the mineralized 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 mineralization models using stratigraphic and SMZ coding to create domains that honour both the vertical and lateral controls on mineralization. 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 (localized Domain 50 and Domain 60 in the far North excluded)



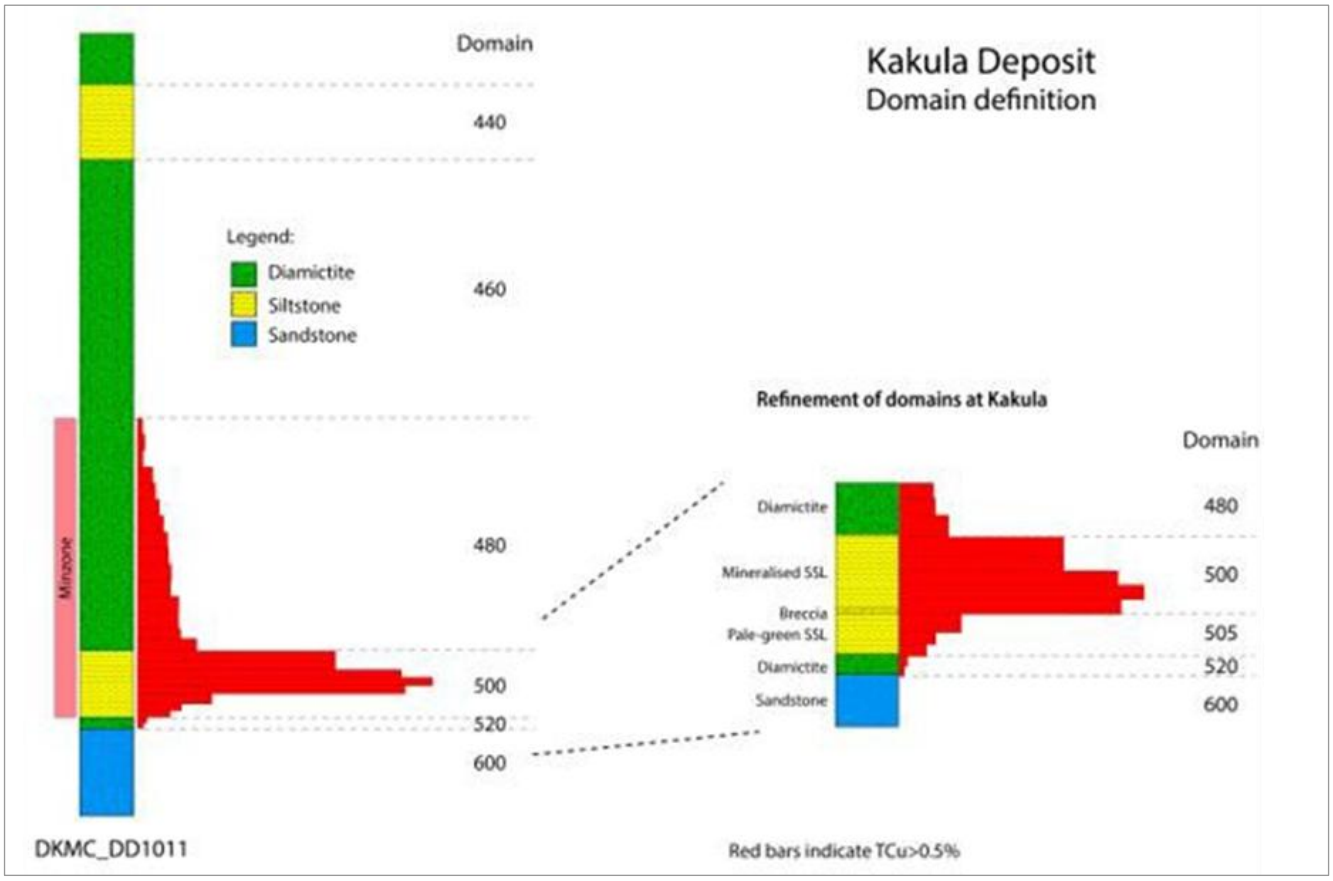
Source: Ivanhoe, 2020.

14.3.2 Kakula

At Kakula, the individual lithological units were combined with the mineralized 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 orientations of the search ranges were adjusted locally using dynamic anisotropy.

Figure 14.2 Kakula vertical domain definition



Source: Ivanhoe, 2023.
Note: 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, honoring 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 amounts of 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 mineralizing event at Kamoa, where the controlling east-west growth fault structure allowed oxidized, copper-rich brines to bypass the lower redox interface at the Roan-Nguba contact and instead access the overlying, highly-sulphidic and reduced KPS. The new, upper mineralized zone hosted in the KPS is characterized 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

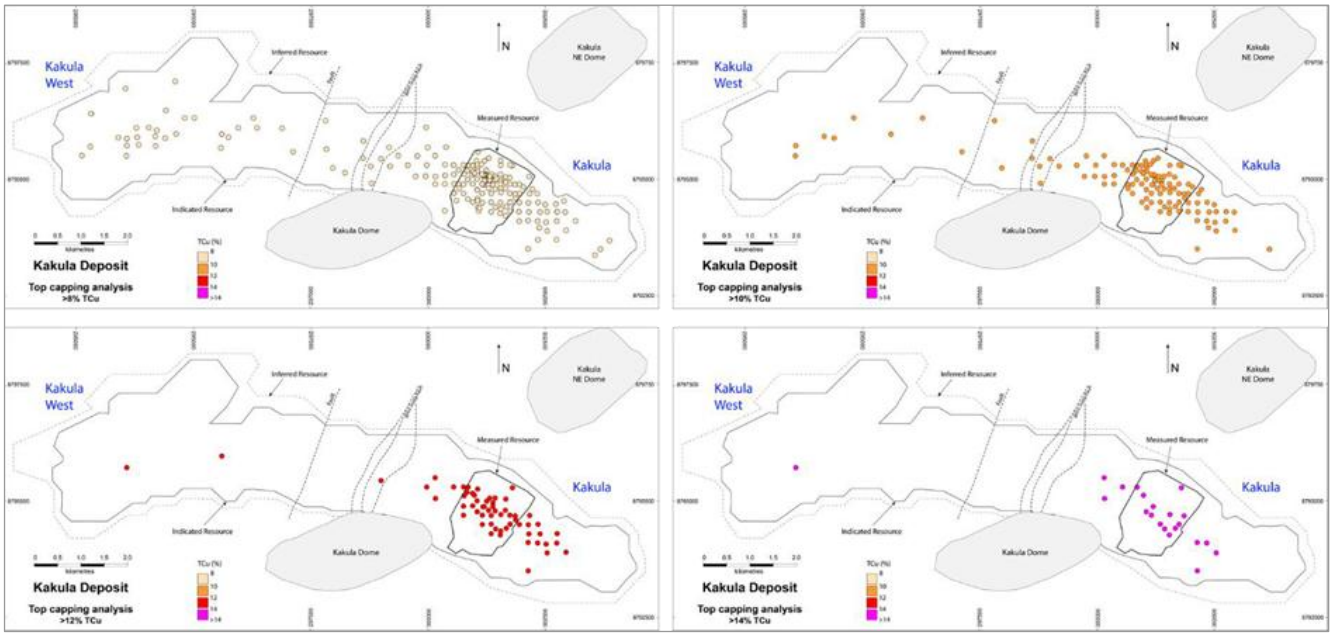
Domain	Number of samples	Capping grade TCu (%)	Samples capped	No capping		With capping	
				Mean (%)	CV	Mean (%)	CV
100	19,927	3.0	12	0.05	3.86	0.05	2.51
110	653	15.0	4	2.51	0.94	2.50	0.92
120	609	35.0	6	7.90	0.86	7.87	0.85
200	8,623	2.6	14	0.22	1.44	0.22	1.42
300	4,627	18.0	6	2.70	0.82	2.69	0.81
400	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 mineralized zone to assess if isolated high-grade samples exist, and whether these values should be capped to prevent over-estimation.

Kakula is characterized 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 mineralized 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 Figure 14.3.

Figure 14.3 Kakula: Visual top capping analyses with TCu grades>8%, >10%, 12%, and >14%



Source: Ivanhoe, 2023.

Table 14.2 Kakula: Impact of top capping per domain on 1 m composite samples

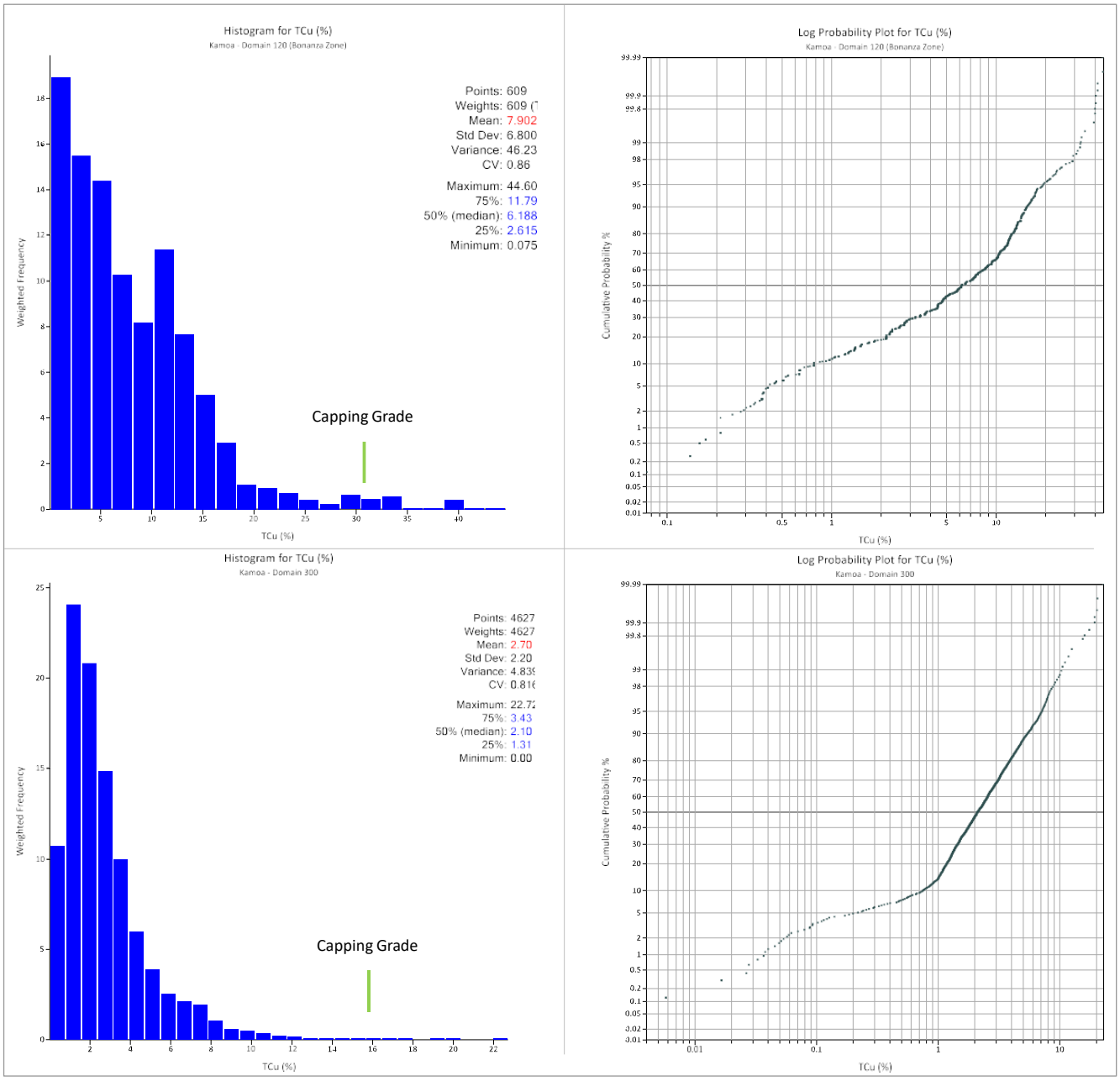
Domain	Number of samples	Capping grade TCu (%)	Samples capped	No capping		With capping	
				Mean (%)	CV	Mean (%)	CV
460	11,288	3.0	5	0.24	1.35	0.24	1.32
480	4,612	8.5	6	1.73	0.71	1.72	0.70
505	316	5.5	4	2.06	0.54	2.04	0.50
520	1,742	3.5	15	0.60	1.20	0.57	0.95
600	2,418	3.0	13	0.14	5.08	0.12	2.82

14.5 Exploratory Data Analysis (EDA)

14.5.1 Kamoa

The distribution of TCu grades within the mineralized 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) & 300 (bottom)



Source: Ivanhoe, 2020.
Note: Green lines represent the top capping applied per domain.

SG values show very little variability, with distributions approximating normal. 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, which is towards the upper end of the diamictite range, likely due to the denser sulphide mineralization. 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.

Sulphur grades are elevated in the KPS due to high concentrations of pyrite within the siltstone. Sulphur grades are also elevated in the mineralized domains, where chalcopyrite dominates. A variety of sulphide species occur within Domain 300 with bornite and chalcocite lowering the overall sulphur grade. Domain 500 is chalcocite-dominant, hence the lower sulphur grades.

Overall, sulphur values are positively skewed.

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

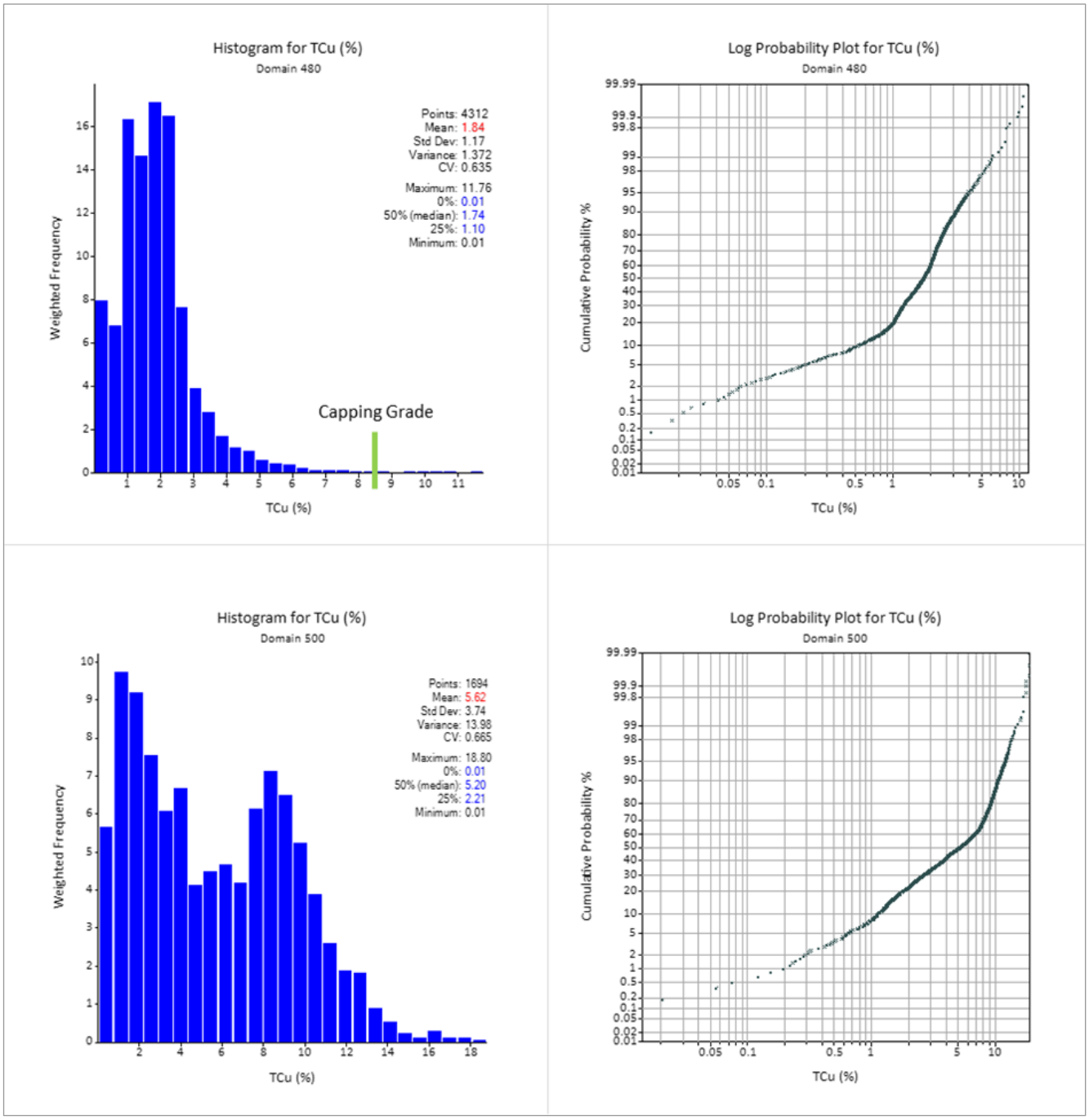
No clear relationship is evident between TCu and ASCu. Higher ASCu grades are usually highly localized and concentrated in only one or two drillholes, indicating an inability to distinguish sulphide and oxide mineralization into separate domains. Geological and metallurgical studies of the sulphide species indicate that the bulk of the mineralization at Kamoa is sulphide, with localized oxide mineralization 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 sulphides where a small amount will dissolve in sulphuric 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 the 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 recognized 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 that was used to assign an SG value to those samples with missing SG values.

Figure 14.5 Kakula: 1 m composite TCu (%) for the mineralization diamictite (Domain 480) and the mineralized portions of the basal siltstone (Domain 500). Histogram & probability plot.



Source: 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 widely-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 mineralized 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 mineralized 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 unoxidized mineralized 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

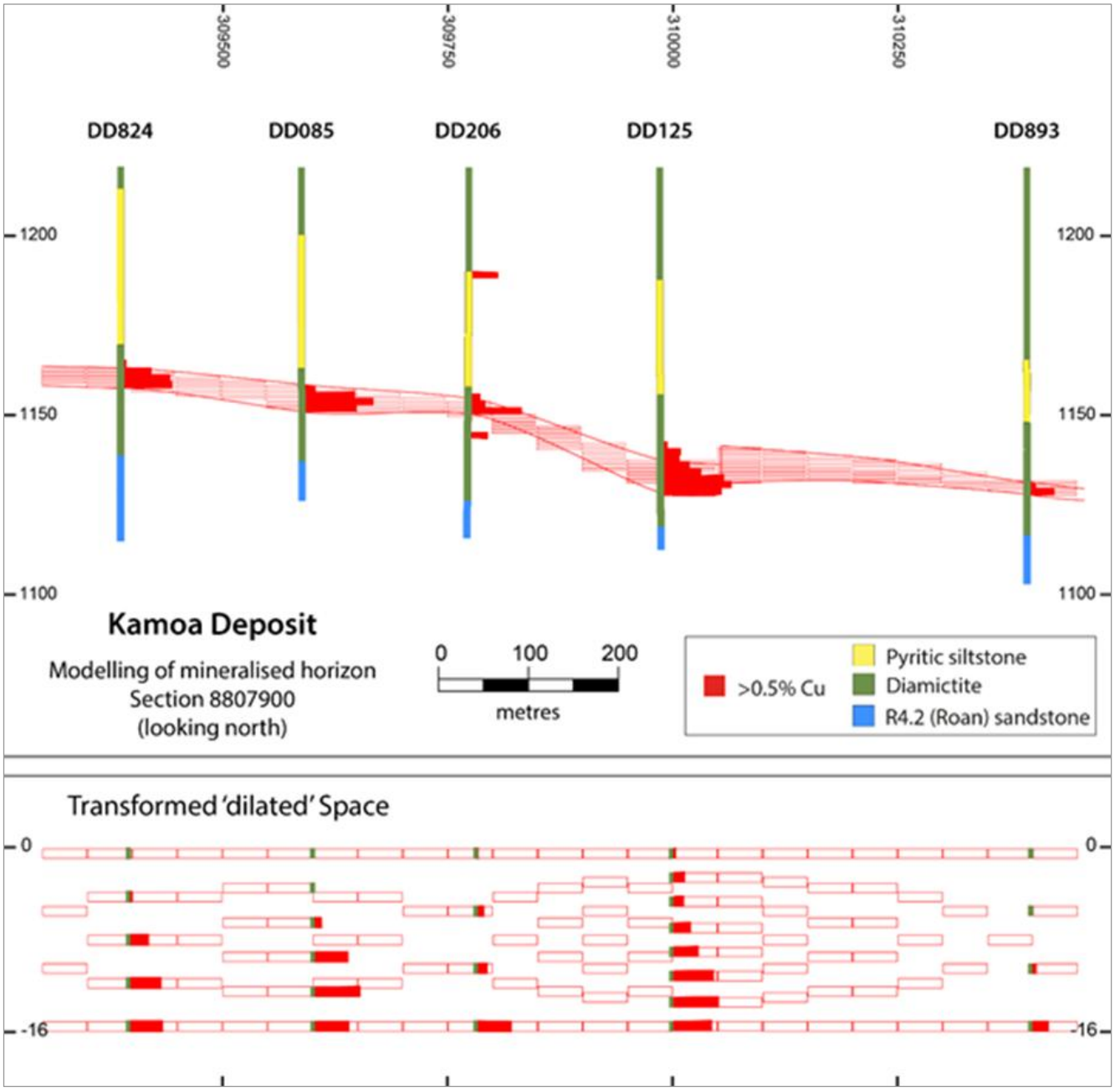
To improve stationarity for grade estimation, both the drillholes and the block model were transformed (“dilated”) to ensure that the vertical TCu grade profiles match between drillholes. Typically, these profiles

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 mineralization 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.

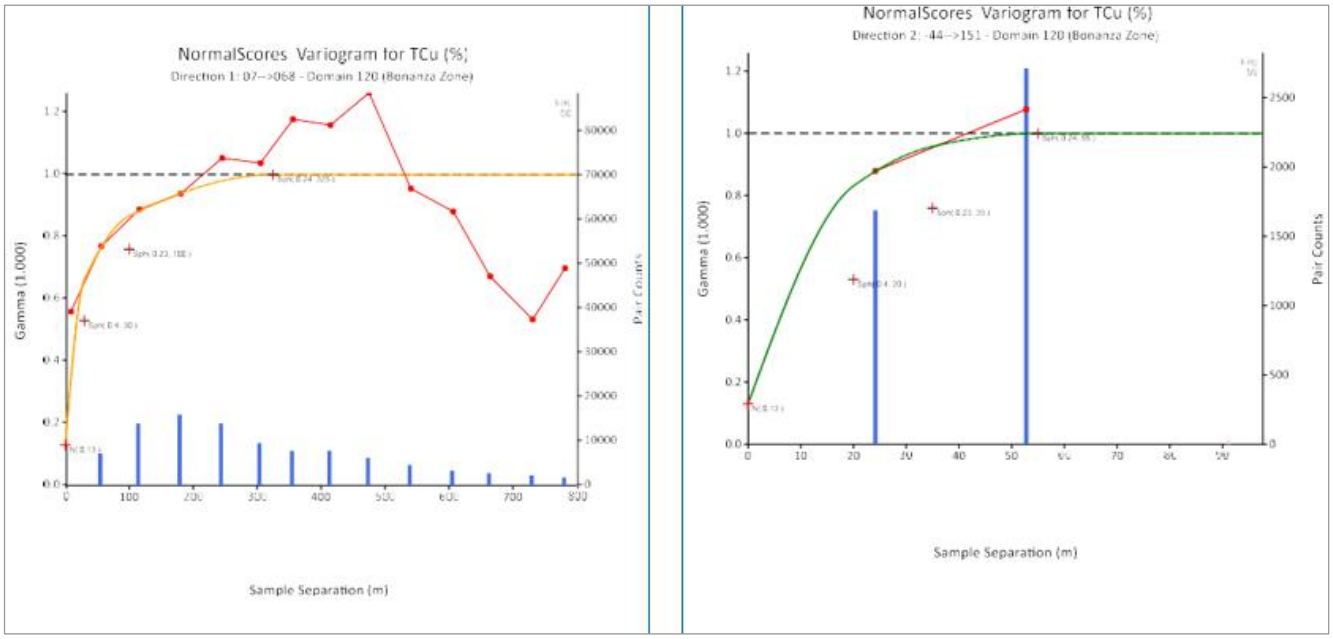
Transformed 1 m composites were used for variography. The variogram parameters were first optimized by performing sensitivity studies on the lag, angular tolerance, bandwidth and a normal-score transform prior to modelling of the variogram. The vertical bandwidth was a key parameter to preserve the vertical TCu grade profiles in drillholes and was typically set to a narrow interval. Downhole variograms of the transformed 1 m composites were used to determine the nugget effect. The transformation expands downhole samples, moving them further away from each other and potentially overstating continuity at short ranges. As a validation, downhole variograms of untransformed 1 m composite samples were also investigated and were found to be comparable. No elevation transform was applied to the Bonanza Zone, and variograms were modelled from samples in their true coordinate positions. Example TCu variograms for Domain 120 (Bonanza Zone) and Domain 300 (Upper SMZ) are shown in Figure 14.7 and Figure 14.8.

Figure 14.6 Kamoa: Vertical section showing untransformed composites and blocks (top) and transformed composites and blocks (lower) for Domain 300, 3 x vertical exaggeration



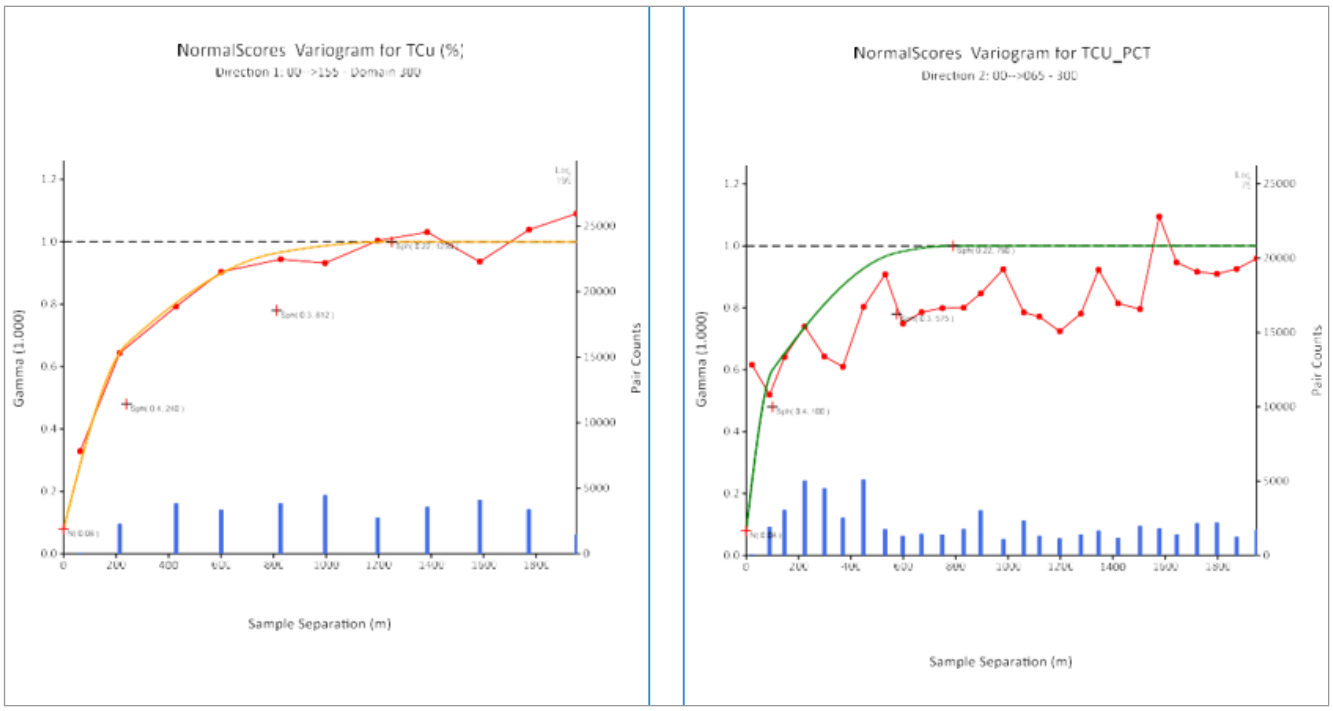
Source: Ivanhoe, 2019.
Note: Copper grade intensity shown by bars on right side of hole.

Figure 14.7 Kamoa: Normal score major and semi-major direction variograms for TCu (Domain 120)



Source: Ivanhoe, 2020.

Figure 14.8 Kamoa: Normal score major and semi-major direction variograms for TCu (Domain 300)



Source: 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 (ID^2) and nearest neighbour (NN) estimates were constructed but were only used for validation purposes. Estimation parameters are summarized 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 dynamic Anisotropy.

Table 14.3 Kamoa: Estimation parameters for TCu for all mineralized domains

Domain	Orientation			Search range	Number of samples		Number of samples	
					Search Pass 1		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

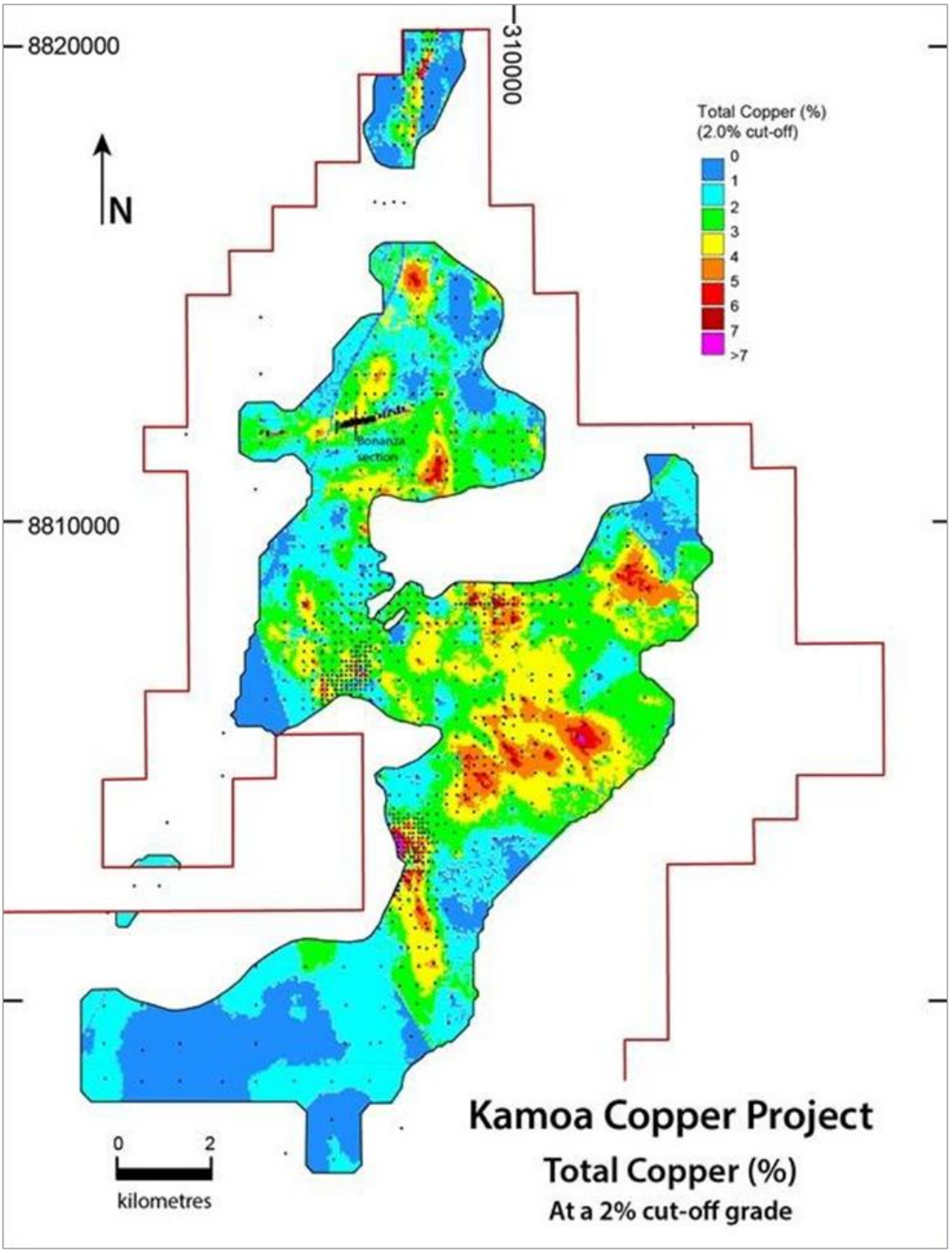
Note: Orientations shown are for overall variography trends; these trends will vary locally as they follow the variable search orientations based on dynamic anisotropy.

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.

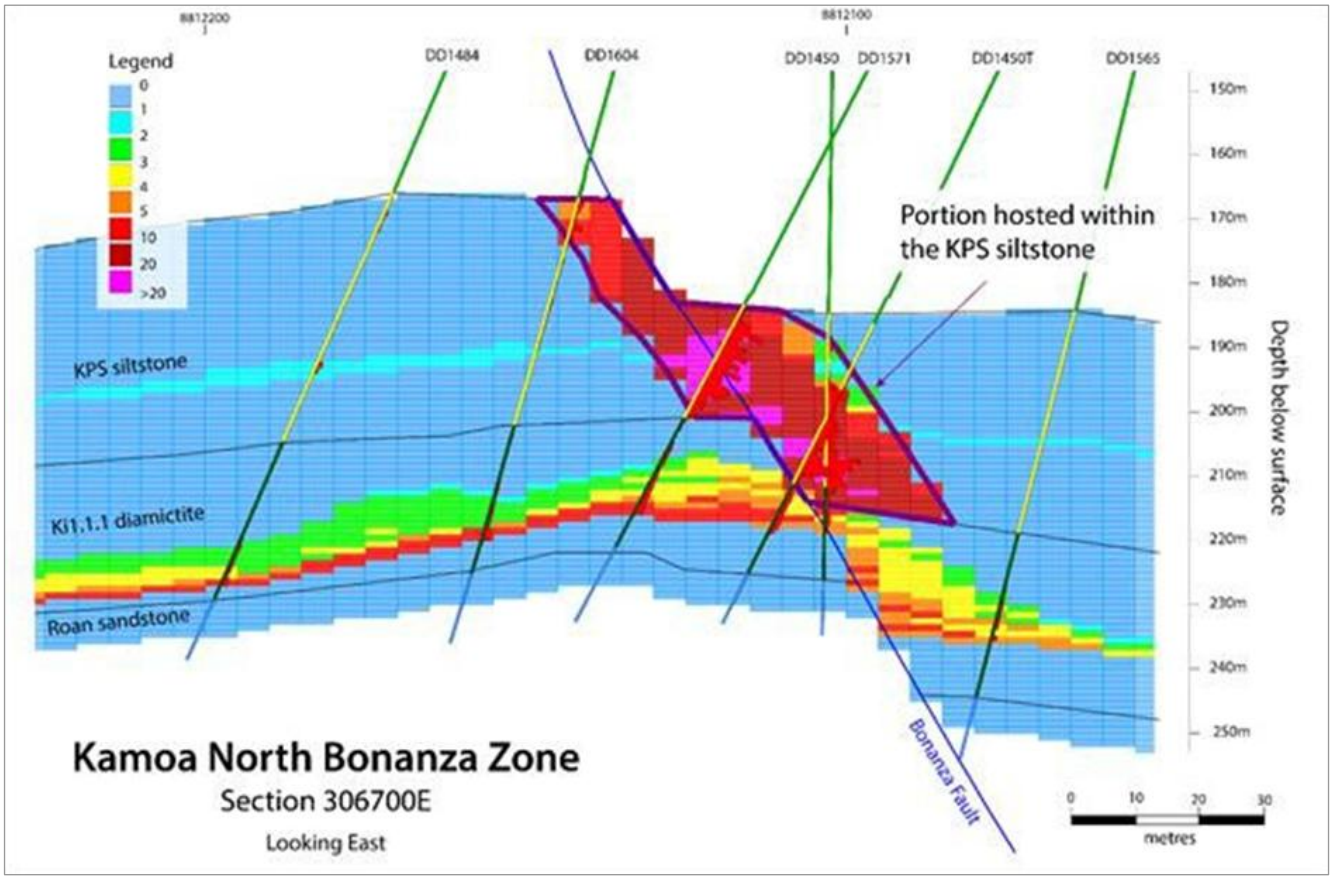
Estimated TCu grades for Kamoa are shown in Figure 14.9. A section view through the Bonanza Zone with estimated TCu grades is shown in Figure 14.10.

Figure 14.9 Plan view of estimated TCu grades at Kamoa (at 2% TCu threshold or grade at minimum 3 m thickness where cut is < 2% TCu)



Source: Ivanhoe, 2020.
Note: 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



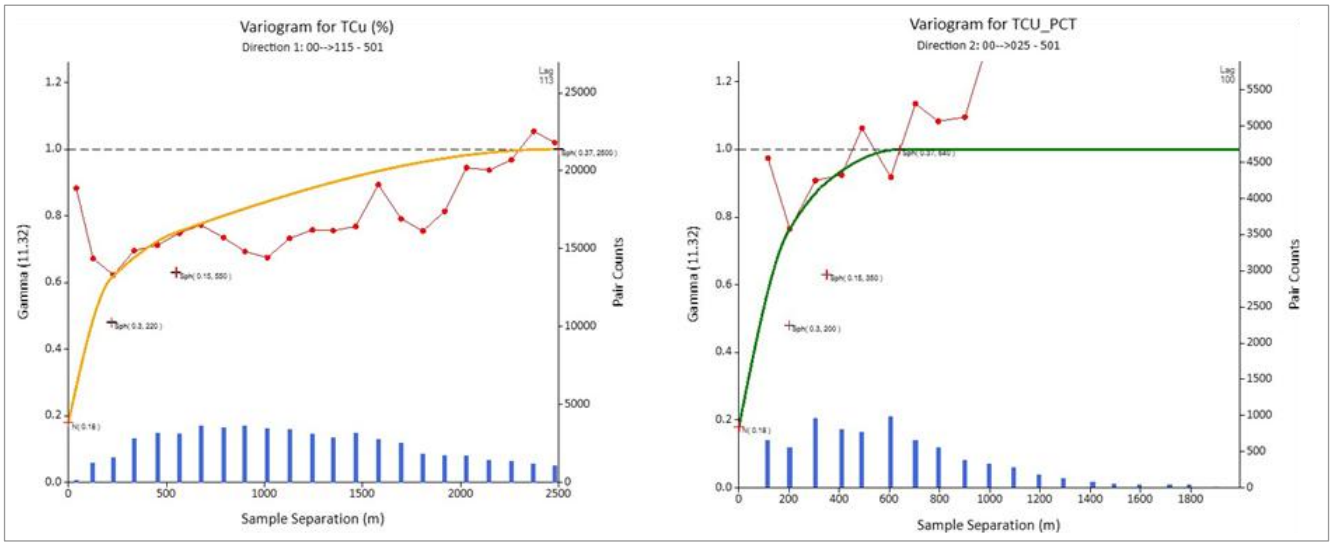
Source: Ivanhoe, 2020.

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 mineralization 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 (mineralized 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 ID^2 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 summarized in Table 14.4.

Figure 14.11 Kakula: Major and semi-major direction variograms for TCu (Domain 500)



Source: Ivanhoe, 2023.

Table 14.4 Kakula: Estimation parameters used for the first search (Domain 500)

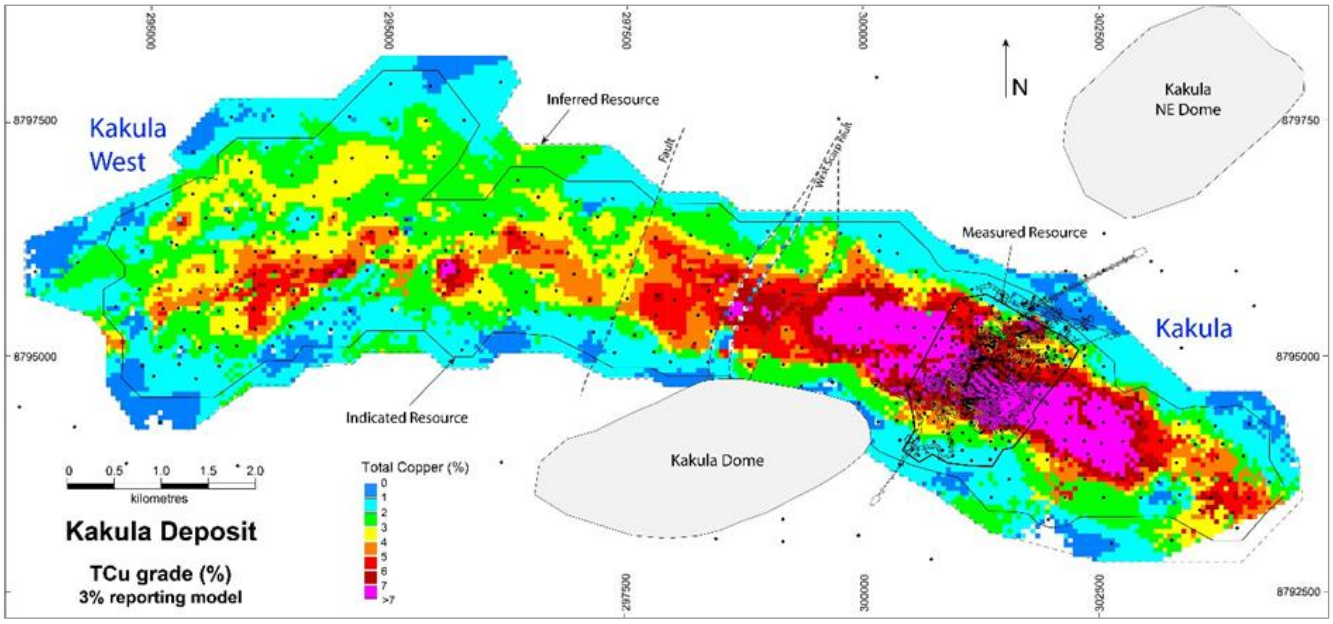
Search domain	Orientation			Search range	Number of samples		Estimation method
	Axis	Azimuth	Dip		Minimum	Maximum	
South-East	X	115°	0°	1,600	4	12	OK
	Y	25°	0°	400	4	12	OK
	Z	0°	-90°	5	4	12	OK
Central	X	110°	0°	1,100	4	12	OK
	Y	20°	0°	600	4	12	OK
	Z	0°	-90°	7	4	12	OK
Western	X	070°	0°	1,500	4	12	OK
	Y	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 recognized 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 to Kakula



Source: Ivanhoe, 2023.
Note: Existing underground development as of 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 ID^2, using only those SG samples that occurred within individual domain wireframes. Search parameters were the same as those used for sulphur.

14.9.2 Kakula

SG data were available for the majority of drillhole samples, and regression values were used where 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 mineralization and robustness of variograms for different domains; modelled continuity ranges far exceed current drillhole spacings used for classification.

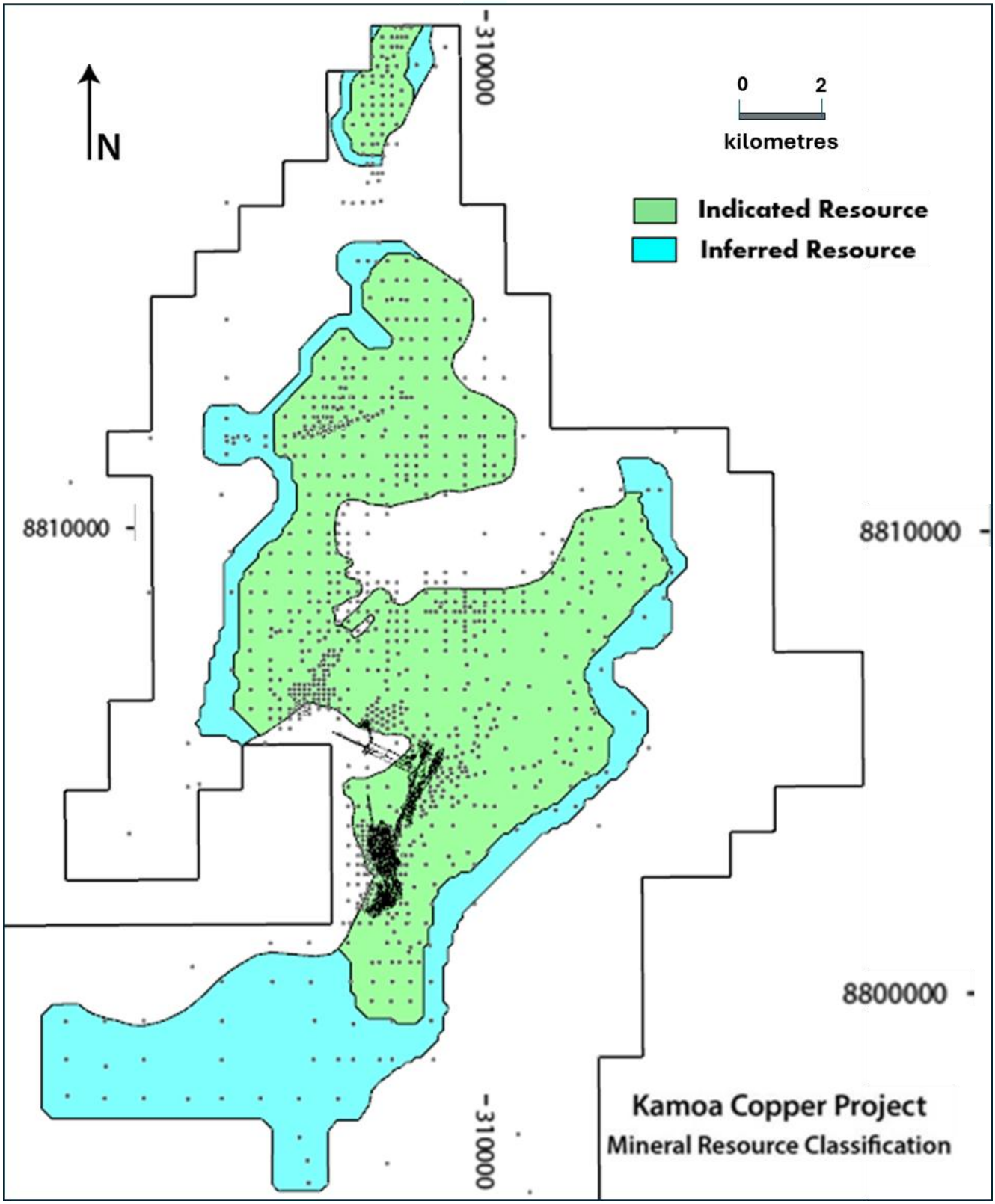
- Comparison of modelled geology units and actual underground exposure.

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. M The total area of the Kamoa-Kakula Project is approximately 397.4 km^2.

The Bonanza Zone represents mineralization 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 mineralization 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 mineralization and account for the additional geological complexity, allowing this zone to be classified as Indicated Mineral Resources.

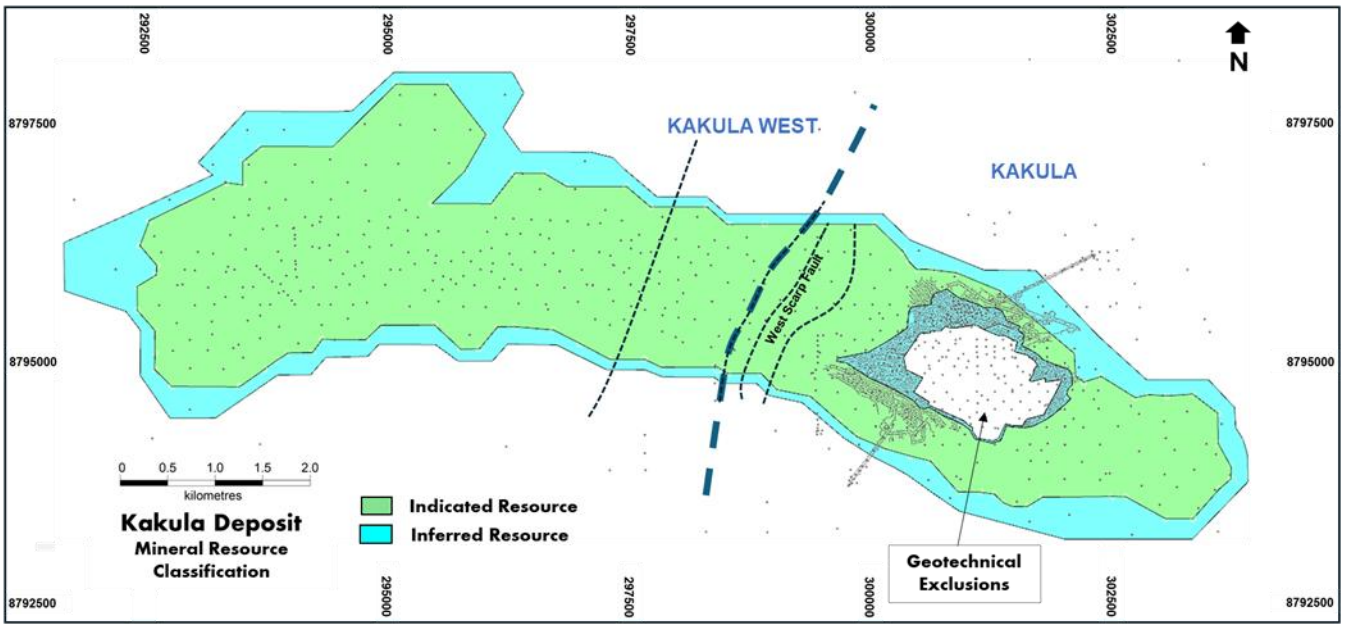
The Mineral Resource classification with drillholes for Kamoa is shown in Figure 14.13, and for Kakula in Figure 14.14.

Figure 14.13 Kamoa Mineral Resource Classification



Source: Ivanhoe, 2020.

Figure 14.14 Kakula Mineral Resource Classification



Source: Ivanhoe, 2026.
Note: Existing underground development as of December 2025.

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: 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 mineralization), and 500 m swaths aligned south-west-north-east (across the trend of the high-grade mineralization). No local biases were evident.

14.12 Reasonable Prospects of Eventual Economic Extraction (RPEEE)

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 mineralization 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 (RPEEE), MSA completed a cut-off grade assessment using reasonably assumed cost and

revenue assumptions based on the following Life-of-mine inputs: copper price $6.00/lb; copper anode produced with payability of 99.7%; concentrator recovery of 86% and smelter recovery of 98.5%; employment of underground mechanized mining methods at a cost of $57.5/t ore mined; concentrator costs of $17.9/t ore milled, general and administrative costs of $21/t of ore milled; smelter costs of $266/t of concentrate; selling cost of $53/t copper produced, royalty of 3.5%, export tax of 1%. Based on these assumptions, the Mineral Resources are considered to have met the requirements for RPEEE at a cut-off grade of 1% copper.

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

To test the cut-off grade for the purposes of assessing Reasonable Prospects of Eventual Economic Extraction (RPEEE), MSA completed a cut-off grade assessment using reasonably assumed cost and revenue assumptions based on the following Life-of-mine inputs: copper price $6.00/lb; copper anode produced with payability of 99.7%; concentrator recovery of 93% and smelter recovery of 98.5%; employment of underground mechanized mining methods at a cost of $57.5/t ore mined; concentrator costs of $23.5/t ore milled, general and administrative costs of $21/t of ore milled; smelter costs of $266/t of concentrate; selling cost of $53/t copper produced, royalty of 3.5%, export tax of 1%. Based on these assumptions, the Mineral Resources are considered to have met the requirements for RPEEE at a cut-off grade of 1% copper.

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.

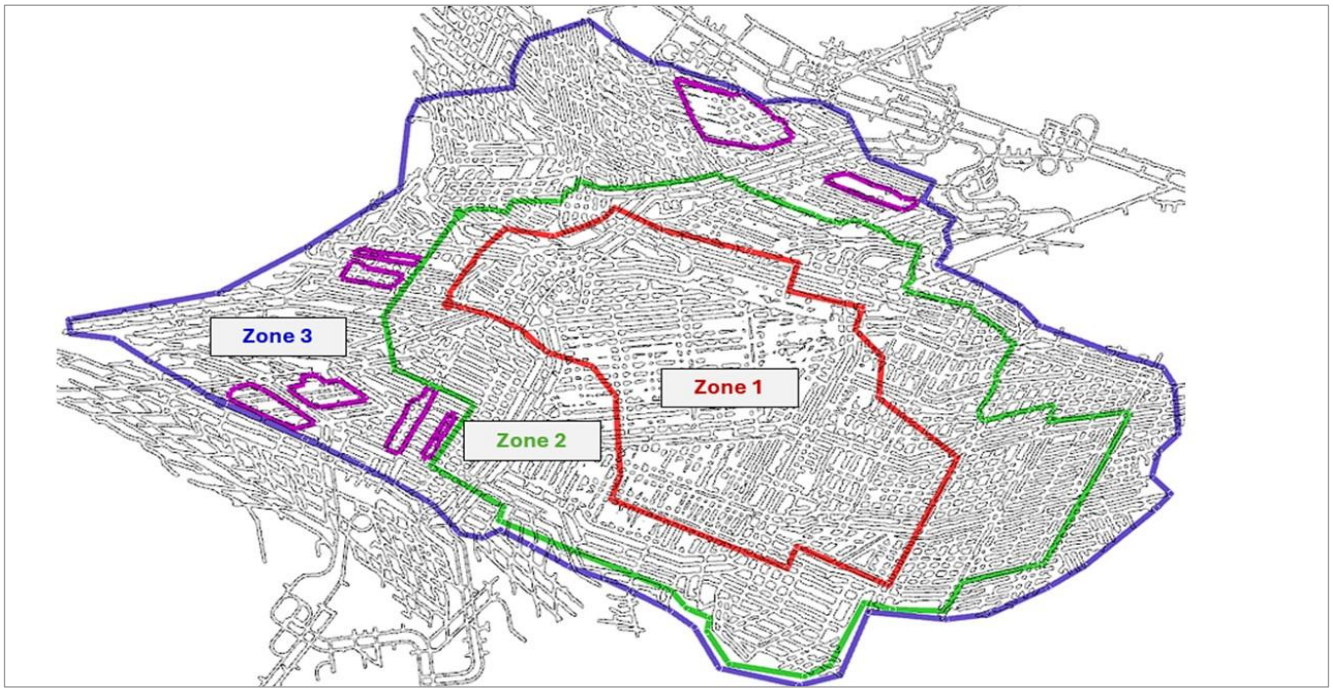
14.13 Mineral Resource depletion

The Mineral Resources have been adjusted to account for mine depletion up to 31 December 2025, as well exclusions of some of Kakula areas affected by seismic activity experienced in 2025. The Kakula depletion is based on actual mined voids. The Kamoa depletion is based on broken-and-trammed ore, with the mined-out voids approach expected to be implemented from 2026 onwards.

14.13.1 Kakula exclusions

Three geotechnical zones related to the degree of seismic activity impact were defined at Kamoa (Figure 14.15). Based on the current geotechnical assessment, Zones 1 and 2 are considered to have exceeded 70% extraction and have therefore been excluded from the declared Mineral Resource. The remaining resource within Zone 3 has been retained but downgraded to the Inferred category, except for areas shown in purple that were interpreted to be effectively mined out and pillars in the areas are excluded from declared Mineral Resource.

Figure 14.15 Indicative - Kakula geotechnical zones



Source: KCSA, 2026

14.14 Mineral Resource statement

Table 14.5 The Kamoa-Kakula 2025 Mineral Resource statement accounts for depletion up to 31 December 2025, as well as geotechnical losses incurred at Kakula deposit during 2025. Some of the Kakula Mineral Resources previously classified as Measured & Indicated have been re-classified into the Inferred and separated into a remnant pillar category for geotechnical reasons and to reflect the lower extraction ratio within this portion of the Mineral Resource, as part of the reasonable prospects for eventual economic extraction (RPEEE), this results in the year-on-year increase in Kakula's Inferred Mineral Resource estimate.

The Mineral Resource was depleted by Joshua Chitambala, B.Min.Sc, MSc, SACNASP (400073/07), Resource Manager, Ivanhoe Mines. An independent review was conducted by Jeremy Witley (Pr.Sci.Nat, SACNASP, FGSSA) of The MSA Group (Pty) Ltd. Both Mr. Chitambala and Mr. Witley are Qualified Persons for Mineral Resources. The effective date of the restated Mineral Resources is 31 December 2025.

The Kamoa and Kakula Mineral Resources are tabulated below.

Table 14.5 Kamoa and Kakula: Mineral Resources (on 100% Project Basis)

Deposit	Category	Tonnage (Mt)	Copper grade (%)	Contained Copper (Mt Cu)
Kamoa	Indicated	750	2.73	21
	Inferred	235	1.7	4.0
Kakula	Indicated	523	2.53	13
	Inferred	75	2.1	1.2
	Inferred Pillars	26	3.5	0.9
Total Kamoa-Kakula Project	**Measured**	**-**	**-**	**-**
	Indicated	**1,272**	**2.65**	**35**
	Inferred	**336**	**1.8**	**6.1**

Notes:

- Mineral Reserves ("reserves") and Mineral Resources ("resources") have been estimated as of 31 December 2025 in accordance with National Instrument 43-101 - Standards of Disclosure for Mineral Projects (NI 43-101) as required by Canadian securities regulatory authorities.
- For 31 December 2025 the long-term copper price used for estimating Mineral Resources is $6/lb.
- 1% total copper (TCu) cut-off grade has been used to report the Mineral Resource.
- Reported Mineral Resources contain no allowances for hanging wall or footwall contact boundary loss and dilution. No mining recovery has been applied.
- The Mineral Resource for Kakula was depleted to account for annual production and losses due to unextractable pillars and inaccessible areas.
- Mineral Resources are reported inclusive of Mineral Reserves.
- Measured and indicated Mineral Resource estimates of grade and proven and probable Mineral Reserve estimates of grade for Cu % are reported to two decimal places.
- All inferred Mineral Resource estimates of grade for Cu % are reported to one decimal place.
- All Mineral Resource estimates of ore tonnes, copper grade and copper tonnes have been rounded to reflect the imprecise nature of the estimates for each classification category, therefore totals may not appear to sum correctly due to rounding.
- Jeremy Witley, Pr.Sci.Nat SACNASP, FGSSA of The MSA Group (Pty) Ltd estimated the Mineral Resources. The 2025 Mineral Resource was estimated from the non-depleted 2023 Mineral Resource estimate, with an effective date of 31 December 2022, and depleted to account for annual production up until 31 December 2025, as well as geotechnical losses incurred during 2025. The 2025 Mineral Resource has an effective date of 31 December 2025.
- The non-depleted 2023 Mineral Resource estimate has an effective date of 31 December 2022 and is documented in the Kamoa-Kakula Technical Report dated 16 March 2023. The cut-off date for drill data at Kamoa is 20 January 2020. The cut-off date for the drill data at Kakula is 20 July 2022, with the assay table updated as of 13 December 2022.
- Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.
- Mineral Resources are reported on a 100% basis. Ivanhoe Mines attributable ownership is 39.6% of Kamoa-Kakula.

14.15 Sensitivity of Mineral Resources to cut-off grade

Table 14.6 summarizes the Kakula Mineral Resource at a range of cut-off grades. The Kakula Measured category has been excluded owing to the difficulty of accounting for remnant ore reclassifications. 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 are not additive to this Table 14.6. Depletion at Kamoa was based on mined-and-trammed ore, making it impossible to accurately determine the grade splits of the remaining resource. The Kamoa sensitivity breakdown has therefore, not been reported.

Table 14.6 Kakula: Sensitivity of Mineral Resources to cut-off grade

Indicated Mineral Resource				
Cut-off (% Cu)	**Tonnage (Mt)**	**Copper (%)**	**Contained Copper (kt)**	**Contained Copper (billion lbs.)**
5.0	57	7.18	4,120	9.1
4.5	67	6.81	4,597	10.1
4.0	80	6.40	5,135	11.3
3.5	98	5.91	5,809	12.8
3.0	124	5.36	6,632	14.6
2.5	166	4.69	7,785	17.2
2.0	246	3.89	9,548	21.0
1.5	354	3.23	11,429	25.2
1.0	**505**	**2.58**	**13,004**	**28.7**
0.5	559	2.45	13,715	30.2
Inferred Mineral Resource				
4.0	2	5.47	86	0.2
3.5	3	4.12	137	0.3
3.0	5	3.51	204	0.4
2.5	9	3.08	315	0.7
2.0	18	2.66	514	1.1
1.5	32	2.20	757	1.7
1.0	**101**	**2.09**	**2110**	**4.7**
0.5	116	1.34	1560	3.4

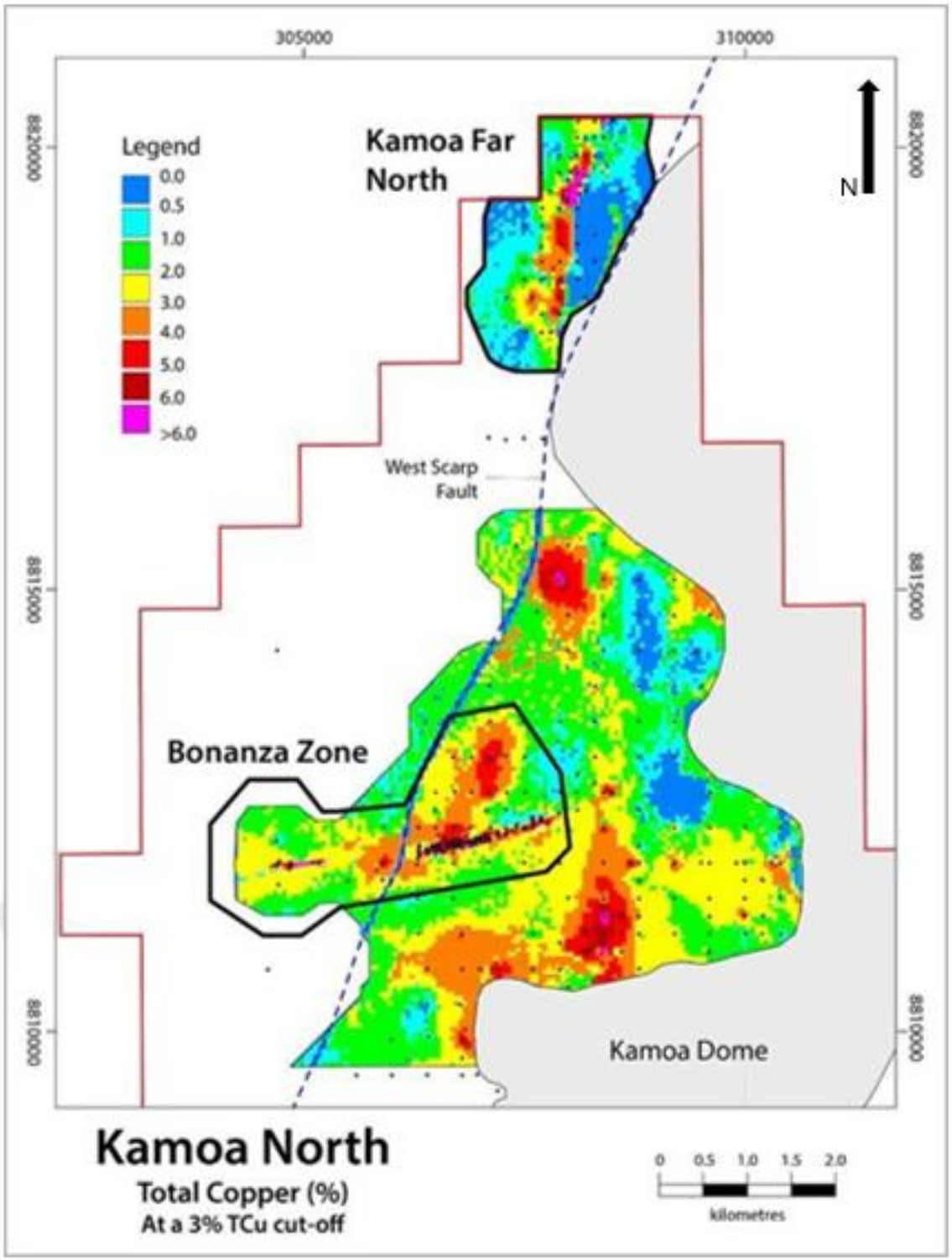
Note: The 1% base case includes geotechnical Zone-3 remnant ore downgraded from Measured category. The footnotes to Table 14.5 also apply to this table. Mineral Resources reported in Table 14.5 are not additive to this table.

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

- Bonanza Zone, incorporating the elevated grades within both the Ki1.1.1 (Domain 300) and KPS-hosted mineralization (Domain 120) (Table 14.7).
- The zone of elevated grade associated with the Bonanza Fault (Domain 120), where mineralizing fluids have had direct access to the highly reducing KPS (Table 14.8).
- Kamoa Far North, in the furthest northern extent of the Mineral Resource on the mining permit (Table 14.9).

Mineral Resources reported in Table 14.5 are not additive to these tables.

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



Source: Ivanhoe, 2020.
Note: 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 14.7 Kamoa Bonanza Zone: Sensitivity of Mineral Resources to cut-off grade

Indicated Mineral Resource					
Cut-off (% Cu)	**Tonnage (Mt)**	**Copper (%)**	**Vertical thickness (m)**	**Contained Copper (kt)**	**Contained Copper (billion lbs.)**
5.0	2	8.89	6.9	212	0.5
4.5	3	7.85	5.8	250	0.6
4.0	4	6.84	5.0	303	0.7
3.5	8	5.53	4.2	421	0.9
3.0	12	4.65	4.1	574	1.3
2.5	20	3.95	4.0	773	1.7
2.0	27	3.50	4.2	933	2.1
1.5	33	3.15	4.4	1,050	2.3
1.0	**37**	**2.95**	**4.5**	**1,100**	**2.4**
0.5	49	2.41	5.4	1,170	2.6
Inferred Mineral Resource					
3.0	1	5.35	4.1	41	0.1
2.5	2	3.84	3.4	72	0.2
2.0	9	2.55	3.8	227	0.5
1.5	16	2.20	4.1	362	0.8
1.0	**19**	**2.09**	**4.3**	**388**	**0.9**
0.5	55	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 14.6, and Table 14.8 are not additive to this table.

Table 14.8 Kamoa Bonanza Zone hosted within the KPS (Domain 120): Sensitivity of Mineral Resources to cut-off grade

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

Indicated Mineral Resource					
Cut-off (% Cu)	**Tonnage (Mt)**	**Copper (%)**	**Vertical thickness (m)**	**Contained Copper (kt)**	**Contained Copper (billion lbs.)**
1.0	**0.5**	**6.24**	**5.2**	**32**	**0.1**
0.5	0.5	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, and Table 14.6 are not additive to this table.

Table 14.9 Kamoa Far North: Sensitivity of Mineral Resources to cut-off grade

Indicated Mineral Resource					
Cut-off (% Cu)	**Tonnage (Mt)**	**Copper (%)**	**Vertical Thickness (m)**	**Contained Copper (kt)**	**Contained Copper (billion lbs.)**
5.0	1	7.17	4.0	78	0.2
4.5	1	6.58	3.9	94	0.2
4.0	2	5.69	4.0	133	0.3
3.5	3	5.09	4.0	171	0.4
3.0	5	4.49	4.0	222	0.5
2.5	7	3.92	4.2	287	0.6
2.0	11	3.39	4.5	365	0.8
1.5	15	2.97	4.9	432	1.0
1.0	**18**	**2.65**	**4.7**	**473**	**1.0**
0.5	24	2.15	4.7	517	1.1
Inferred Mineral Resource					
1.5	0.2	1.97	2.7	5	0.0
1.0	**2**	**1.32**	**2.9**	**21**	**0.0**
0.5	6	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, and Table 14.6 are not additive to this table.

14.16 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 kilometers and a vertical mineralization extent of a few metres that include small scale structures which influence the mine planning and the physical mining. These challenges, however, are minimized by the significant amount of high-quality drillhole data and the general consistency and predictability of the mineralization.

The 3D modelling method was designed to provide the flexibility to adjust the mining height or grade profile on a local scale to optimize the mine plan and potentially improve the Project economics.

14.17 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.

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 mineralization.

14.18 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 prefeasibility level mine planning.

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

- Drill spacing:
  - The drill spacing at the Kamoa and Kakula deposits is insufficient to determine the effects of local faulting on lithology and grade continuity assumptions. Local faulting and steep dips around growth faults can disrupt productivity.
  - 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. Mineralization at Kakula appears to be more continuous compared to Kamoa.
- Assumptions used to generate the data for consideration of RPEEE 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 RPEEE 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 test work 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 mineralization 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 mineralization currently identified at Kakula that requires a dedicated recovery model separate from the hypogene recovery prediction method.
- Commodity prices and exchange rates.
  - These will fluctuate over the life of the project, which may impact future assessments of cut-off grade and RPEEE.
- Cut-off grades:
  - Cut-off grades are impacted by production and sales costs, metallurgical recoveries, commodity prices, and exchange rates, all of which are assumed for the purposes of Mineral Resource declaration.

- Mineral Resources in and around Zone 1 & Zone 2 of Kakula Mine:
  - Portions of the area outside of the mature extraction zone and estimated barrier pillar that have not reached approximately 70% extraction remain in the Mineral Resource statement. While it is assumed that some of the area will be extractable, and therefore Mineral Resources remain in the area for possible mine planning, it remains uncertain what proportion can be extracted.

15 Mineral Reserve estimates

The Mineral Reserve estimate for the Kamoa-Kakula mining complex was reviewed and approved by Karl van Olden, Global Lead –Underground Mining, AMC Consultants, a Qualified Person as defined under National Instrument 43-101. The effective date of the Mineral Reserve estimate is 31 December 2025.

The Mineral Reserve estimate is shown in Table 15.1. This Mineral Reserve estimate is current at 31 December 2025. The Kamoa-Kakula Mineral Reserve is reported on a 100% basis.

Mineral Reserves are reported in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019).

The estimate uses economic and physical modifying factors, the latest Mineral Resource and geological models (as described in Section 14), geotechnical and hydro-geological inputs, and metallurgical processing recovery relationships. The Mineral Reserve is supported by work completed to a prefeasibility study (PFS) level of confidence, consistent with NI 43-101 requirements for Mineral Reserve reporting. All deposits are supported by mine plans at a minimum PFS level of confidence, as described in Section 16.

Mine designs incorporate the geotechnical guidelines developed following the review of the May 2025 geotechnical event at the Kakula mine, including revised block span dimensions, inter-block pillar requirements, backfill sequencing controls and extraction ratio limitations, as described in Section 16.

Mineral Reserves have been estimated for the portions of the Kakula deposit that are unaffected by the geotechnical event in May 2025 only. Areas affected by the May 2025 geotechnical event have been excluded from the Mineral Reserve estimate pending completion of detailed underground rehabilitation and mine design studies sufficient to demonstrate safe and economically viable extraction.

Mining cut-off grade scenarios were evaluated between 1.5%TCu and 2.0%TCu. The cut-off grade applied to each individual mining area has been based on economic cut-off grade, practical mining parameters, Mineral Resource and geology characteristics and spatial considerations. Mineral Reserves have been evaluated using a long-term copper price of US$4.50/lb which the Qualified Person considers reasonable for the purposes of demonstrating economic viability. As described in Section 21, all costs used to assess economic viability yare expressed in Q1 2026 US dollars. No escalation has been applied.

Mineral Reserves incorporate mining dilution and mining recovery factors applied during mine design. Dilution factors range from 0% to 20% and mining recovery factors range from 85% to 95%, varying by mining method and deposit, as described in Section 16. Pillar recovery at the end of mine life has been assumed at 30% of total pillar inventory, subject to prevailing geotechnical conditions at the time of extraction.

Metallurgical recovery varies by deposit and by ore type. Recoveries applied are based on test work results as described in Section 13.

Mineral Reserves are reported on a dry tonne basis, and grades are reported as total copper (%TCu) as delivered to the primary, run of mine crushing facilities.

Table 15.1 Kamoa-Kakula Complex - Mineral Reserves Summary, 31 December 2025 (100% Basis)

Deposit	Proven			Probable			Proven and Probable		
	Ore (Mt)	Copper (%)	Copper (Contained Mt)	Ore (Mt)	Copper (%)	Copper (Contained Mt)	Ore (Mt)	Copper (%)	Copper (Contained Mt)
Kakula	-	-	-	51.3	3.94	2.02	51.3	3.94	2.02
Kakula West	-	-	-	84.4	2.98	2.52	84.4	2.98	2.52
Konsoko Sud	-	-	-	32.6	2.71	0.88	32.6	2.71	0.88
Kamoa 1	-	-	-	103.9	2.71	2.82	103.9	2.71	2.82
Kamoa2	-	-	-	78.0	2.59	2.02	78.0	2.59	2.02
Kamoa 3	-	-	-	57.6	2.41	1.39	57.6	2.41	1.39
Kamoa 4	-	-	-	42.7	2.46	1.05	42.7	2.46	1.05
Kamoa 5	-	-	-	8.6	2.66	0.23	8.6	2.66	0.23
Kamoa 6	-	-	-	7.2	2.74	0.20	7.2	2.74	0.20
Total	**-**	**-**	**-**	**466**	**2.82**	**13.13**	**466**	**2.82**	**13.13**

Notes:

- Mineral Reserves (“reserves”) and Mineral Resources (“resources”) have been estimated as at 31 December 2025 in accordance with National Instrument 43-101 - Standards of Disclosure for Mineral Projects (“NI 43-101”) as required by Canadian securities regulatory authorities.
- For 2025 the long-term copper price used for calculating Mineral Reserves and economic mine plan analysis is $4.50/lb. The long-term copper price used for calculating Mineral Resources is $6.00/lb.
- Realization costs include refining and treatment charges, deductions and payment terms, blister and concentrate transport, metallurgical recoveries, and royalties.
- Cut-off grades applied to the Mineral Reserve are between 2.0% TCu and 1.5% TCu. The varying characteristics of each deposit, and the intention of maintaining reliable mining parameters and geotechnical controls has resulted in each scenario applying both a minimum economic cut-off, practical mining parameters and spatial considerations to differentiate between mined material considered to be ore or waste.
- In confirming the Mineral Reserves for Kamoa & Kakula, a reserve test has been undertaken, to verify that the future undiscounted cash flow from reserves is positive. The cash flow ignores all sunk costs and only considers future operating and closure expenses as well as any future capital costs.
- Metallurgical recovery for each Concentrator is defined by the application of a recovery algorithm. The metallurgical recovery is 87.98% for the Kakula and Kamoa concentrators (Mineral Reserve life-of-mine plan average).
- Smelter recovery is 98.5%.
- Mineral Reserve tonnage and grade estimates include apportionment for dilution and recovery.
- Mineral Reserves reported above are inclusive of Mineral Resources and are not additive.
- Totals may not appear to sum correctly due to rounding.
- All Mineral Resource and Mineral Reserve estimates of tonnes, Cu tonnes and pounds are reported to the second significant digit.
- Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.
- Measured and indicated Mineral Resource estimates of grade and proven and probable Mineral Reserve estimates of grade for Cu % are reported to two decimal places.
- All inferred Mineral Resource estimates of grade for Cu % are reported to one decimal place.

16 Mining methods

16.1 Introduction

The Kamoa-Kakula mining complex, operated by KCSA comprises nine distinct underground deposits. An overview of the Kamoa-Kakula deposits is shown Figure 16.1. The Kamoa 1 to 6 deposits are located in the central to northern extents of the property. Kansoko Sud is centrally located on the property and the Kakula and Kakula West deposits are located to the south. This report separates the Kakula West deposit into two regions, Kakula West-East and -West.

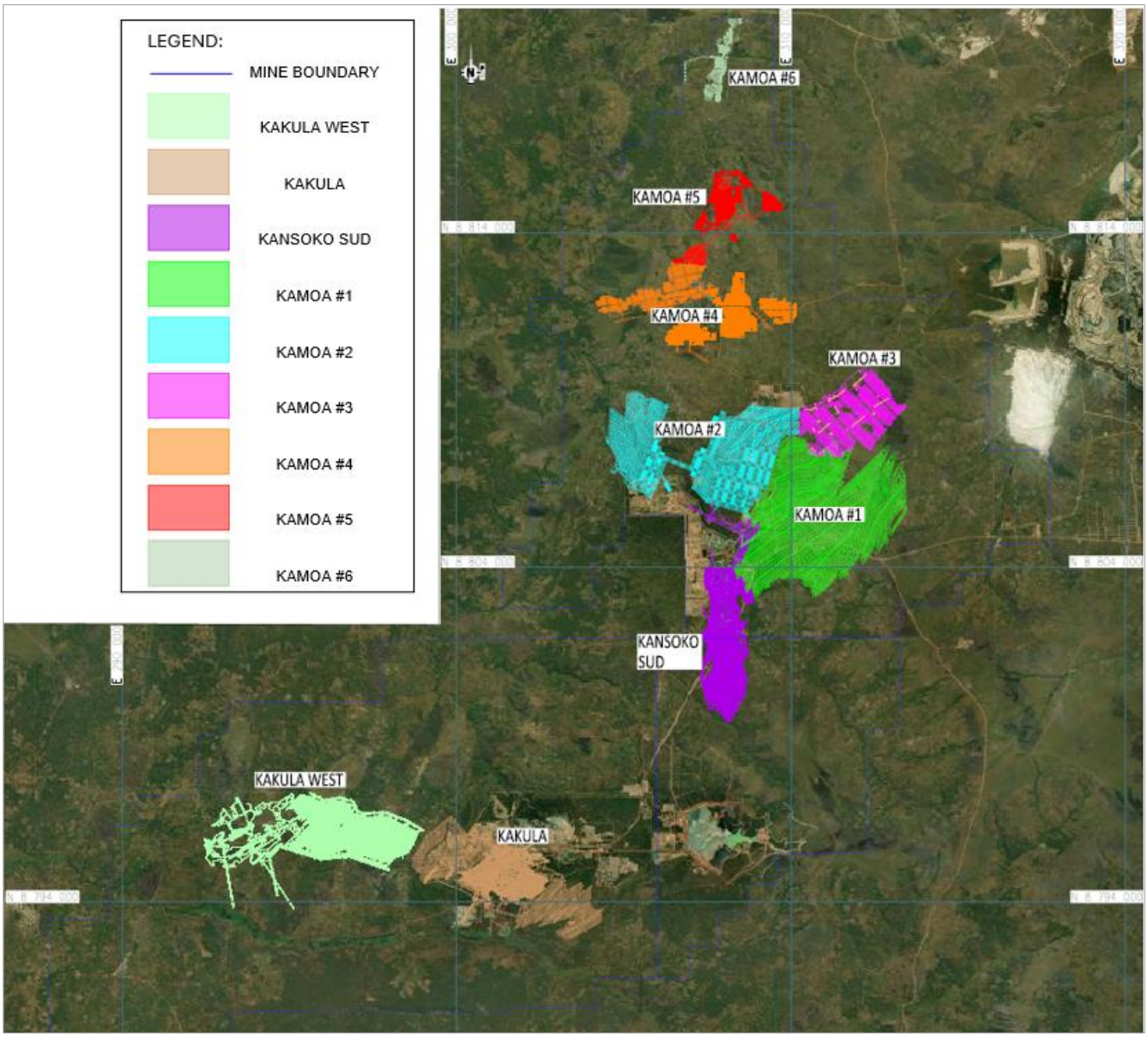
The Kamoa concentrator (Phase 3) is located adjacent to Kamoa 1 and Kamoa 2. The Kakula concentrator (Phase 1 and 2) and Smelter are located adjacent to the Kakula deposit. The property extends approximately 30 km north to south and between 30 to 20 km east to west.

In 2026, active mining operations are underway at the Kakula, Kamoa 1, Kamoa 2 and Kansoko Sud operations. The mine planning that underpins the Mineral Reserve Estimate for the operating mines is guided by current life-of-mine plans with appropriate adjustments to ensure the plans meet the requirements of a compliant Mineral Reserve estimate. The Mineral Reserve estimate for the remaining deposits is based on updated studies using current resource, design, cost, infrastructure and other relevant parameters. All the planning and study work used to estimate Mineral Reserves is at least at a prefeasibility study (PFS) level of confidence. KCSA will continue to investigate opportunities to improve the value of the mineral assets and increase the productivity and profitability of these mines.

The life-of-mine sequence and schedule used in this Mineral Reserve plan aligns with the current production philosophy of KCSA. Strategic studies are underway to further explore optimization of significant value drivers such as:

- Materials handling of ore from underground to surface and to the concentrators.
- Development sequence of each deposit.
- Production rates from each deposit.
- Dynamic cut-off grades applied across the life of each deposit.
- Dewatering and depressurization of the Roan Sandstone.

Figure 16.1 Kamoa-Kakula site plan



Source: KCSA 2026

16.2 Mining district geotechnical overview

16.2.1 Deposit setting

The underground mines exploit stratiform orebodies with dips ranging from approximately 9° to 35°, averaging 18°. Orebody thickness varies between 2.5 m and 20 m, with a typical average of 6 m. The deposits occur across a range of depths, from surface outcrop to 1,400 m below ground level (mBGL), with planned mining extents spanning from shallow workings at 70 mBGL to 1,400 mBGL. Current operations (2026) are approaching depths of approximately 300 mBGL.

Ore mineralization is hosted within layered sedimentary sequences comprising breccias, siltstones, and diamictite. Across the mining footprint, the orebody exhibits undulating geometry with displacement

amplitudes of 0.5 m to 10 m, which presents challenges for on-reef mining continuity and necessitates frequent small-scale adjustments to both dip and strike orientation.

16.2.2 Geotechnical setting

The geotechnical review for the Kamoa-Kakula 2025 PFS was undertaken by AMC Consultants Pty Ltd (AMC) with input and regional stability guidance from Beck Engineering Pty Ltd (Beck), Mining 3 Pty Ltd (Villaescusa) and Open House Management Solutions (OHMS). Previous work was undertaken by SRK Consulting (South Africa) Pty Ltd (SRK) and has been used as a guide where no new information was available.

AMC has carried out various geotechnical reviews and input into 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, where available 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 mining methods including room and pillar, drift-and-fill and long hole stoping (LHS), which includes sliping (see Section 16.2).

Note: Geotechnical assessments and recommendations generic to all operations are summarized in this section (16.2.5) and are not repeated in the individual sections below. Locally specific geotechnical information is listed in the individual sections where relevant.

16.2.3 Geotechnical data

The geotechnical assessments for Kakula West, Kamoa 3, Kamoa 4, Kamoa 5 and Kamoa 6 were based on the following available information:

- Beck (2025) Technical report: Initial Appreciation of a Damaging Event at Kakula Mine, Prepared by Beck Engineering, May 2025.
- Beck (2025) Analysis of a large- scale instability event at Kakula Mine, October 2025.
- Kamoa Copper SA (2023) Kakula Lab Rock Properties. Excel spreadsheet, 2023.
- Kamoa Copper SA (2023) Kamoa Lab Rock Properties. Excel spreadsheet, 2023.
- Kamoa Copper SA (2024) Updated Underground Support Units Design and Requirements, 20 July 2024.
- Kamoa Copper SA (not dated) West Skarp Fault. Mine planning considerations before connecting the Kakula Central and Kakula West mines.
- Kamoa Copper SA (2025) All Kamoa Copper Support Standards Book. 18 February 2025.
- Kamoa Copper SA (2025) Changes to mine design: Kamoa Copper SA. Report KCK0001_Geotechnical Assessment, Rock Engineering Department, 6 May 2025.
- Kamoa Copper SA (2025) KCSA LOM Plan 17 Mtpa. Presentation, 10 June 2025.
- Kamoa Copper SA (2025) GCD TARP June 2025. Excel spreadsheet, 12 June 2025.
- Kamoa Copper SA (2025) Support Standard identification at Kakula. 14 July 2025.
- Kamoa Copper SA (2025) Kamoa-Kakula Geology presentation. Frank Twite,12 August 2025.

- Kamoa Copper SA (2025) Life-of-mine Plan 2026. 13 November 2025.
- OHMS (2024) Kamoa and Kansoko Operations – Life-of-mine Numerical Modelling Assessment, 21 December 2024.
- OHMS (2025) Pillar Stability Assessment for Kakula As-mined. Open House Management Solutions PowerPoint Presentation, May 2025.
- OHMS (2025) Kakula Operation Quarterly Numerical Modelling Assessment, 31 May 2025.
- OHMS (2025) Kamoa Kansoko Operations Quarterly Numerical Modelling Assessment, 31 May 2025.
- OHMS (2025) Kamoa Copper Kakula Operation Failure Investigation, 30 June 2025.
- OHMS (2025) Kamoa Copper Future Projects, Gap Analysis. Open House Management Solutions PowerPoint Presentation, August 2025.
- OreWin (2020) Technical report: Kamoa-Kakula Development Plan 2020, October 2020.
- SRK (2020) Feasibility level underground Geotechnical Investigation and Design for Kakula Mine. Report 541087 prepared by SRK Consulting, February 2020.
- SRK (2022) Kamoa-Kakula Phase 3 UG PFS Geotechnical Investigation and Design. Prepared by SRK Consulting (Report number 586019. November 2022.
- WSP (2025) Kamoa Prefeasibility Update for NI-43-101 Update. Reference 41108009-V01, August 2025.
- Villaescusa (2025) Geotechnical Review of Large-Scale Instability Kakula Copper Mine, Democratic Republic of Congo. Mining 3, November 2025.

OHMS, 2025 completed a gap analysis for recommended future study work based on Target Level of Data Confidence (TLDC) as shown in Table 16.1. Recommendations for further geotechnical data collection to achieve the appropriate level TLDC are provided for each mine footprint.

Table 16.1 TLDC for underground mining excavations for rock mass properties (after Grenon, et al., 2015)

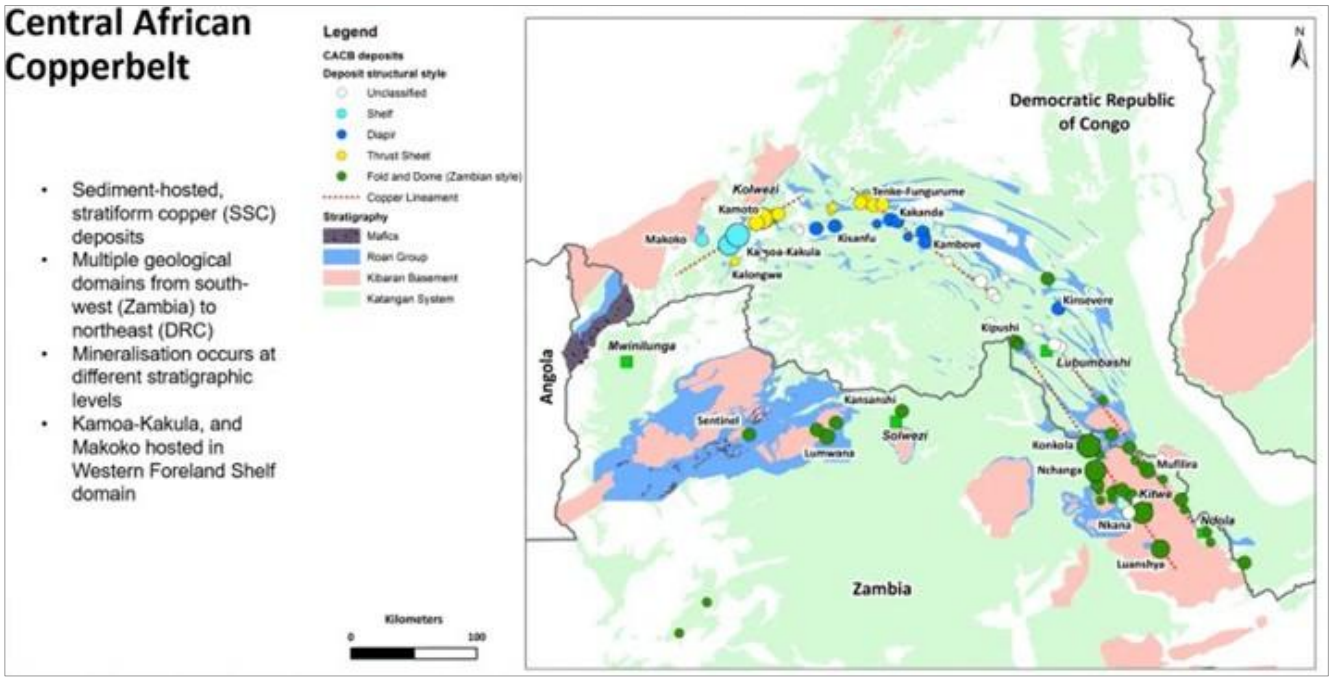
Project level status	Conceptual	Pre-Feasibility	Feasibility	Design and Construction	Operation	
					Temporary	Permanent
Geotechnical level status	Level 1	Level 2	Level 3	Level 4	Level 5	Level 6
Rock mass TLDC (%)	> 30	40 – 65	60 - 75	70 - 80	80 - 85	> 85
Er (%)	**< 70**	**35 - 60**	**25 - 40**	**20 - 30**	**15 - 20**	**< 15**

Limited additional geotechnical information has been collected since 2020. AMC has reviewed the information available from previous studies for determined rock mass characteristics, RQD, RMR and Q-system values, along with a review of available core photographs and Kamoa Copper Future Projects Gap Analysis undertaken by OHMS, 2025. This has guided recommendations on the proposed mining methods and ground support recommendations.

16.2.4 Geotechnical domains

The geotechnical characteristics are influenced by the three primary lithological domains, Siltstone (SST), diamictite (SDT) and sandstone (SST), from the Roan basement rock, across all deposits. The Roan Group is shown in blue in Figure 16.2 below.

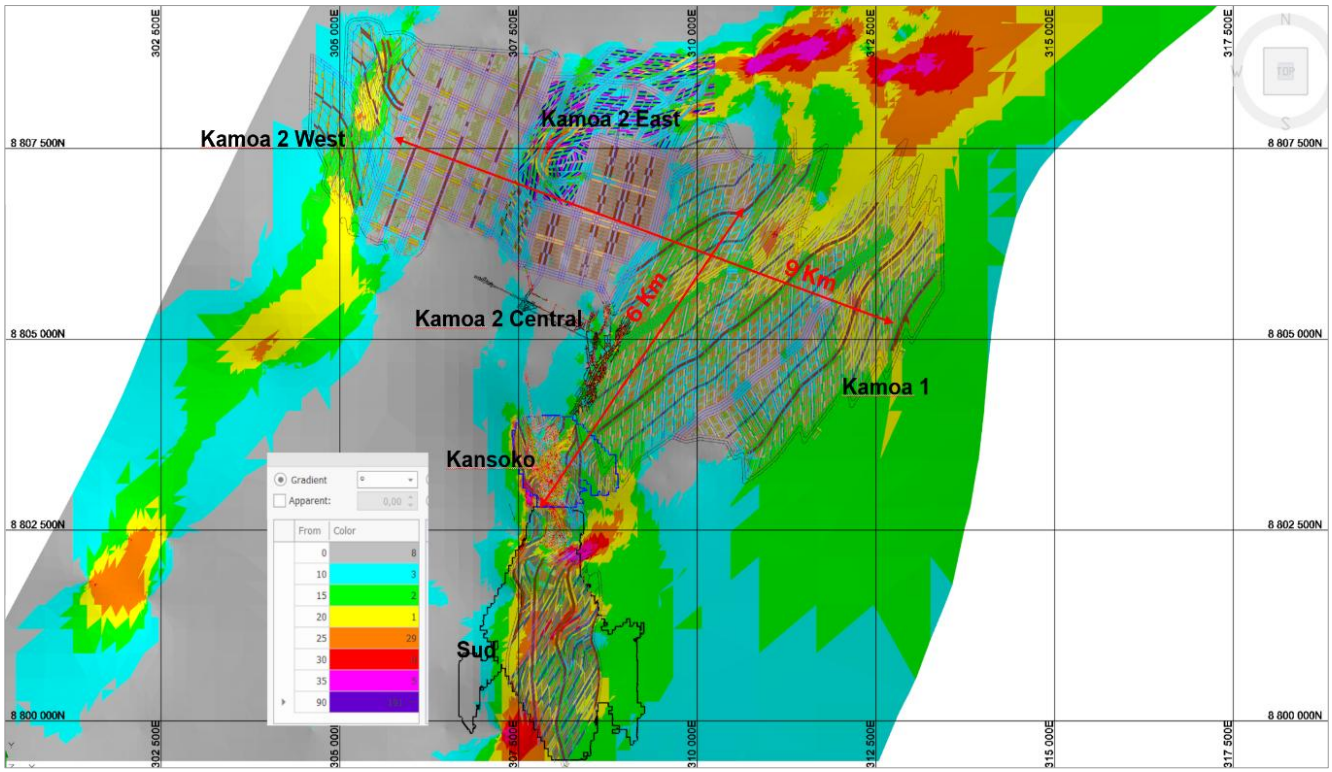
Figure 16.2 Central African Copperbelt



Source: KCSA 2025

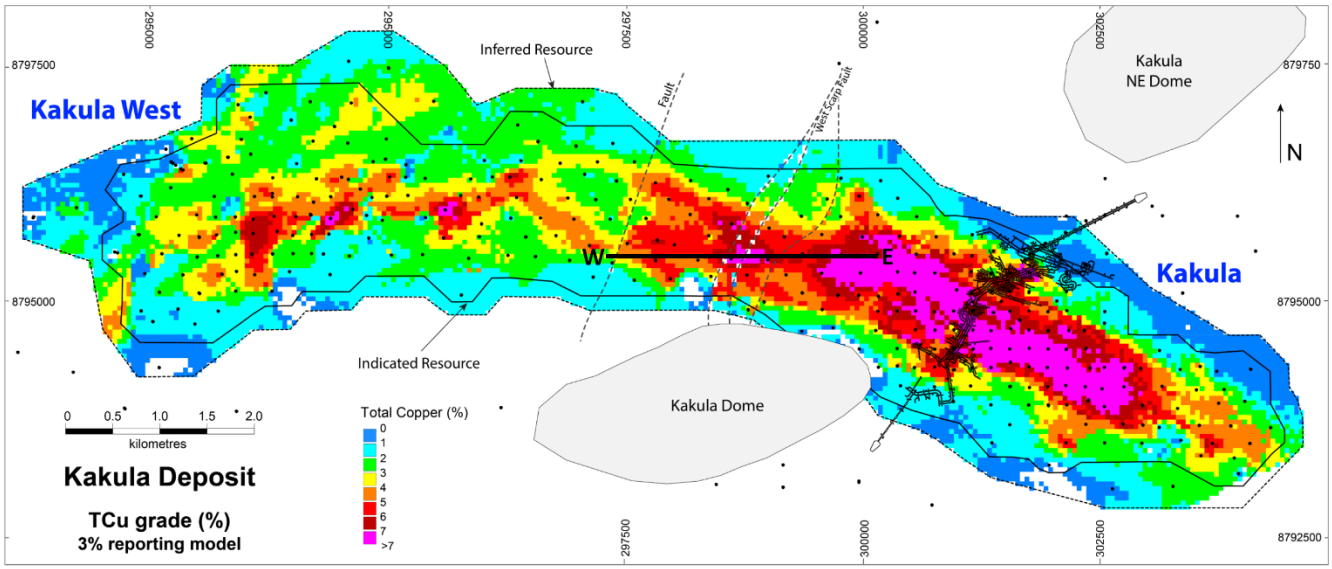
The Roan Group stratigraphy forms domes with the deposits dipping shallow to steeply away from the domes. There is a significant difference in the orebody deposition between the Kamoa and Kakula sites. Variable rock mass conditions can be attributed to the domes, and regional rift graben structures that offset, and major faults including the West Skarp and Bonanza faults. Grade distributions are shown for Kamoa in Figure 16.3 and Kakula-Kakula West where the link between primary structures and grade distribution is illustrated in Figure 16.3.

Figure 16.3 Kamoa and Kansoko deposits with dip analysis



Source: KCSA, 2025

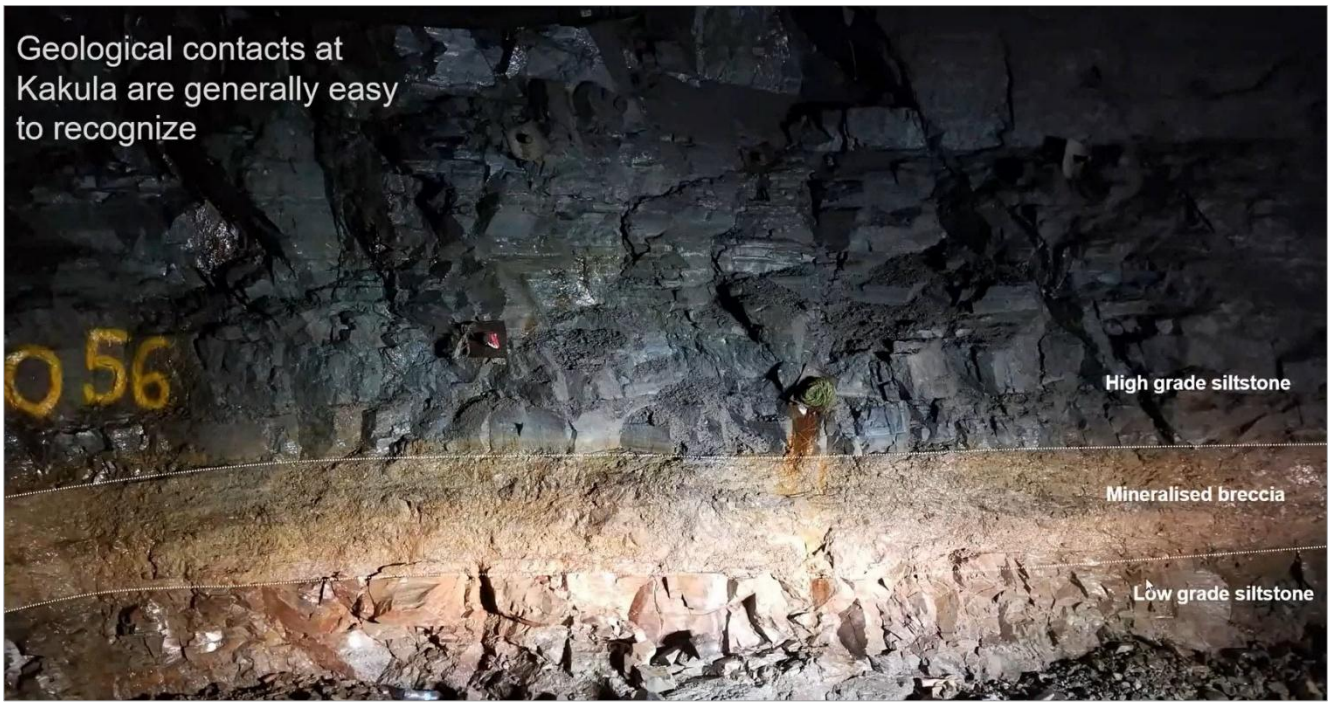
Figure 16.4 Kakula deposit



Source: Ivanhoe, 2025

At Kakula the geological contacts are generally easy to recognize as illustrated in Figure 16.5. Experience at Kakula has shown that the breccia becomes critically unstable when exposed over a long mining front, which result in sheared pillars and tunnel damage.

Figure 16.5 Kakula geological contacts



Source: KCSA 2025

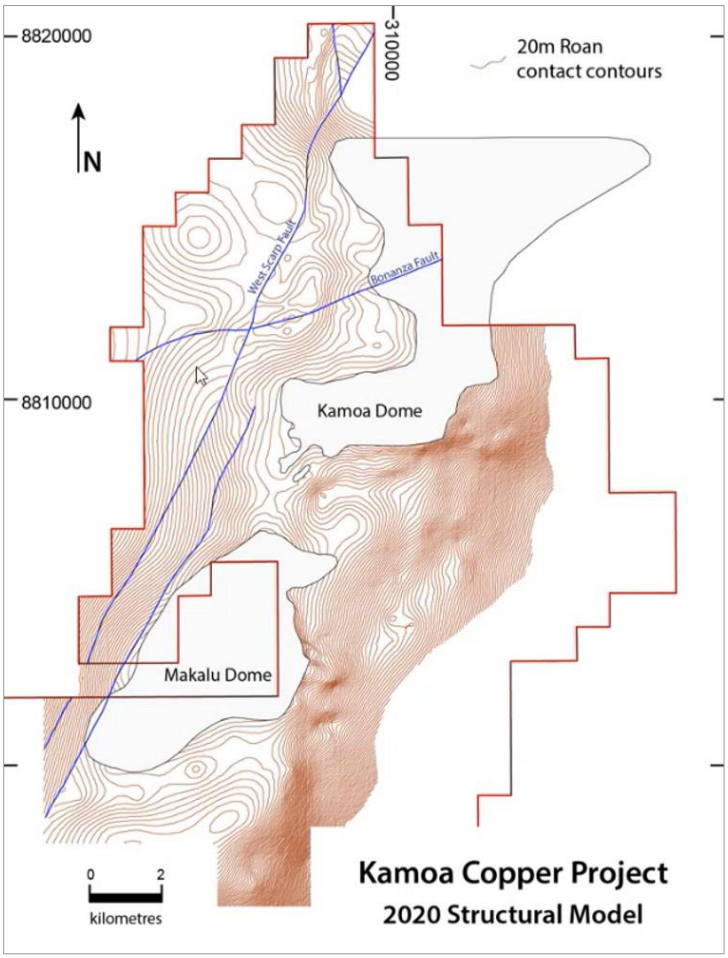
Identifying the variability of the local geology is often limited due to the wide drillhole spacing and a challenge to mine planning and scheduling until the development stage drilling and investigation is completed. This is due to the basin architecture and rift system often shown as localized orebody offsets or folding, often within short distances some 20 m to 30 m.

The orebody offsets, or primary depositional faults, which are roughly oriented northwest in Kamoa and west to northwest at Kakula, can be difficult to find in drilling because they occur in very short distances, and whilst are not always associated with poor or damaged rock mass conditions. However, there is a link between the primary depositional faults for grade distribution and thickness. The Kamoa and Kakula-Kakula West deposit structural models and Roan sandstone contact contours are shown in plan (Figure 16.6 and Figure 16.7) and in cross section in Figure 16.8, where offset in the deposit can be seen to range from 50 m to 250 m.

The Bonanza Zone and West Skarp Fault are characterized as tectonic faults with wide zones of poor ground that are highly jointed, broken or brecciated. Drillhole DD1080 in Figure 16.9 provides an example of the rock quality of the West Skarp Fault with 60 m indicated to be of badly broken ground.

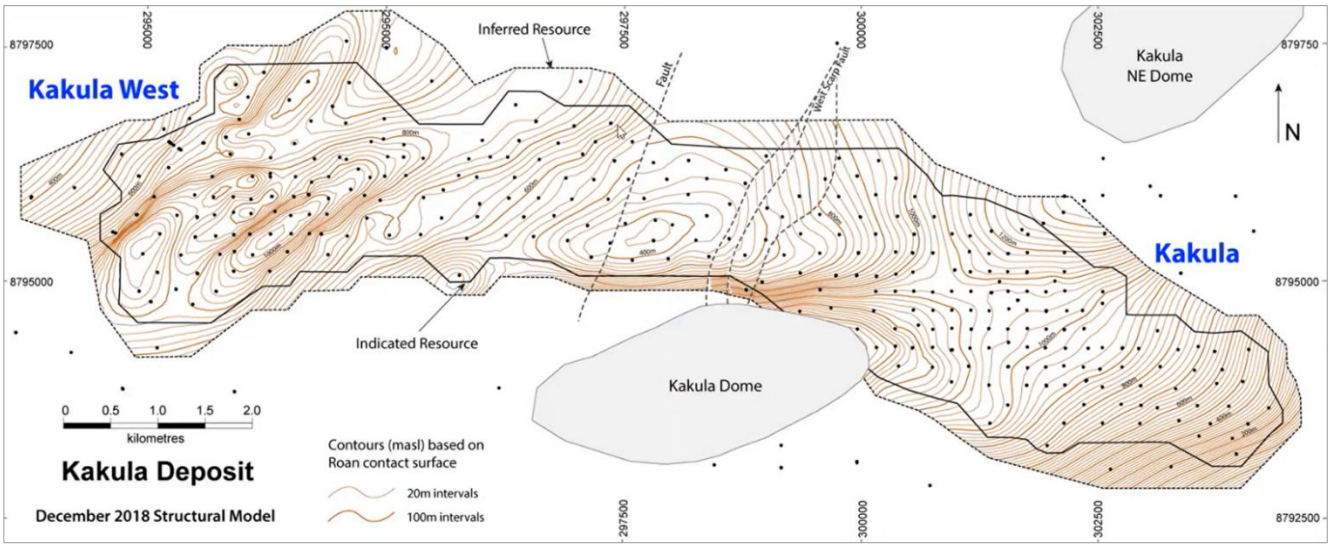
Weaker rock mass conditions can be attributed to soft sediment folding. The folding example in Figure 16.10 shows soft sediment deformation as well as flat dipping siltstone.

Figure 16.6 Kamoa deposit with structural model



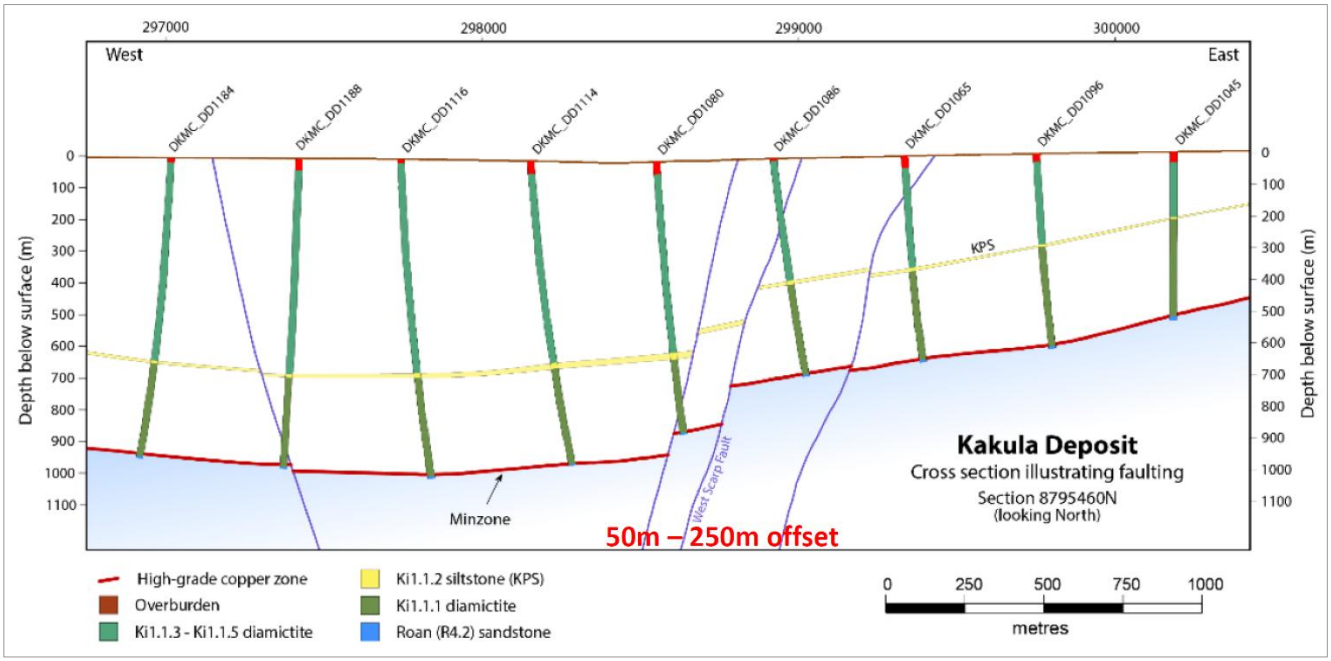
Source: KCSA 2025

Figure 16.7 Kakula – Kakula West deposit with structural model



Source: KCSA 2025

Figure 16.8 Cross section Kakula – Kakula West deposit with structural model



Source: KCSA 2025

Figure 16.9 Kakula - Kakula West deposit – West Skarp Fault Rock quality DD1080

Source: KCSA 2025

Figure 16.10 Example of localized folding

Source: KCSA 2025

16.2.5 Geotechnical assessment

16.2.5.1 Rock mass quality assessment

Rock mass classification allows the relative integrity of the rock mass to be quantified based on the assessment of several key characteristics measured or observed from drill core or rock exposures. Empirical relationships to excavation stability and support requirements have been developed based on rock mass classification, and these are used (in the preliminary stages) for various mining-related geotechnical applications.

Rock mass characterization considered the following three main rock types:

- Diamictite (SDT)
- Siltstone (SSL)
- Sandstone (SST)

Based on the assessment of rock material strength, RQD, discontinuity spacing and condition, the different rock domains have been classified using Rock Mass Rating (RMR) system (Bieniawski, 1989). Geological Strength Index (GSI), estimated from the RMR values, is used to estimate rock mass shear strength parameters. Hoek, Kaiser and Bawden (Hoek et al, 1995) provide descriptions of these classification systems and their inter-relationship. Rock Tunnelling Quality Index, Q, (commonly referred to as the 'Q-system') after Barton, Lien and Lunde (1974), has application in underground development assessments. These classification systems are considered appropriate for all rock materials.

The characterization of the rock mass is a critical component of the mine design process. Mine stability can be comprised when there are inconsistencies between data sets, gaps or limited quality control for validation of parameters during mining.

Conventional empirical rock mass classification systems previously discussed, used data derived from core logging, photographs and limited laboratory testing in previous studies. These methods have provided a preliminary understanding of the rock mass quality, however ongoing validation is required as mining develops to ensure the complex geological conditions encountered are considered with detailed modelling of mining methods and schedules.

RMR calculations

Rock mass characterization was carried out using RMR_{89} (Bieniawski, 1989). It is calculated as follows:

$$RMR_{89} = A1 + A2 + A3 + A4 + A5$$

The "A" parameters relate to the following:

- Strength of intact rock (Factor A1)
- Drill core quality (RQD) (Factor A2)
- Discontinuity spacing (A3)
- Discontinuity conditions (A4)
- Ground water (A5)

Rock Tunnelling Quality Index (Q)

Rock Tunnelling Quality Index, Q, (commonly referred to as the 'Q-system') after Barton, Lien and Lunde (1974), has application in underground development assessments (Table 16.2). Q is calculated as follows:

$$Q = \frac{RQD}{Jn} \cdot \frac{Jr}{Ja} \cdot \frac{Jw}{SRF}$$

Where:

RQD is the rock quality designation, a rock quality estimation technique developed by Deere et al (1967).

Jn is the joint set number, assigned based on the number of joint sets (values ranging from 0.5 to 20).

Jr is the joint roughness number, assigned based on the shape and roughness of joint surface (values ranging from 0.5 to 4.0).

Ja is the joint alteration number, assigned based on the mineral infill and alteration of the joint surface (values ranging from 0.75 to 20).

Jw is the joint water number, assigned based on the expected water inflow into an excavation (values ranging from 0.05 to 1).

SRF is the stress reduction factor, which modifies Q to account for the effects of in situ stress or presence of fault zones. Values typically range from 0.5 to 20, and up to 400 in high stress, rockbursting conditions.

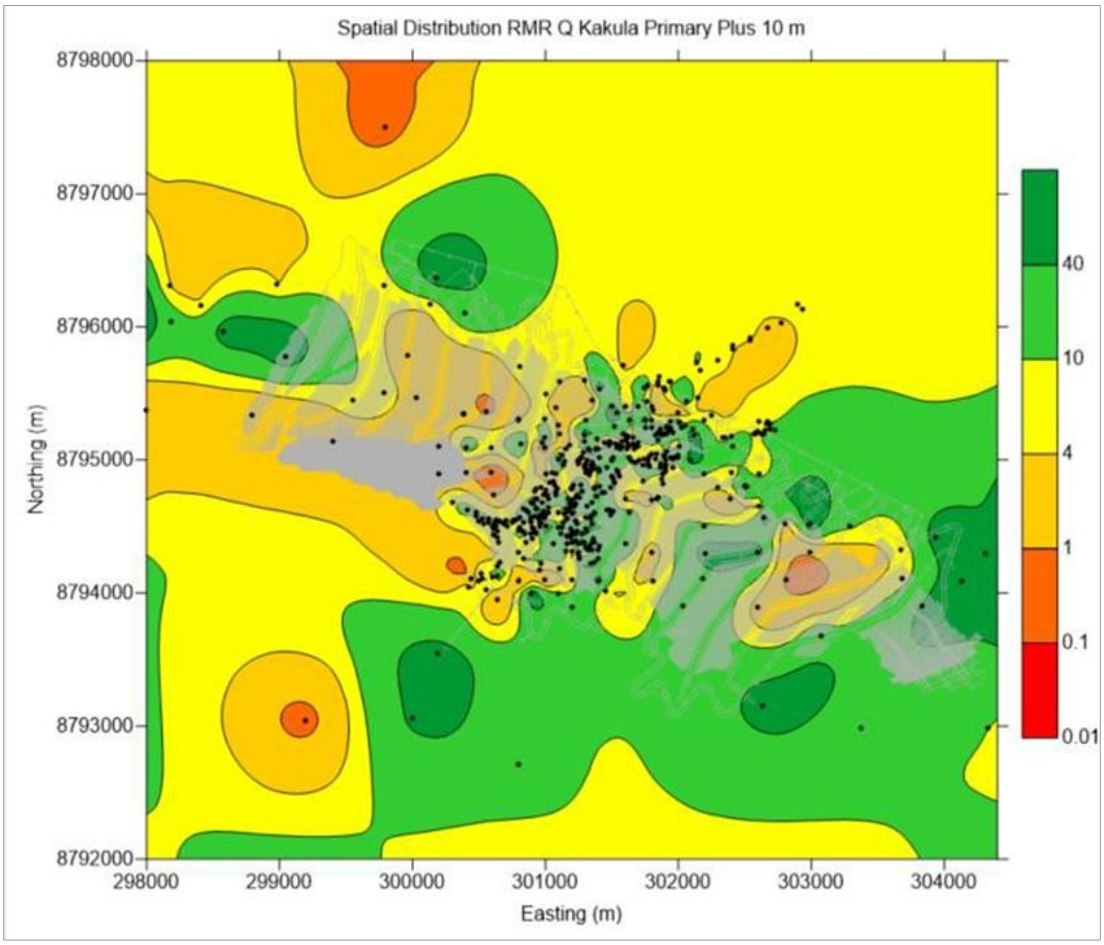
Table 16.2 Rock mass classification based on the Q-system (Barton Lien and Lunde, 1974)

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**

Limited new geotechnical drillhole information was available. The Q-contour maps generated by SRK, 2018 for each mining footprint were a first pass guide for the rock mass rating. The Q contours provided an indication of the different rock mass classes expected within the direct hanging wall and at specific intervals selected above the hanging wall. The contours were developed using a 10 m wireframe interval to allow for variability within the orebody and hanging wall. The evaluation was based on median Q data which was considered representative data of the geotechnical information available.

Previous geotechnical evaluation for five mining footprints was based on geotechnical drilling, logging, and underground mapping where available by Ivanhoe Mines over the Kamoa project area, which was reviewed and interpreted into Q contour maps by SRK, 2018, considering 5 m hangingwall rock mass and 6 m orebody thickness (Figure 16.11) and essentially unchanged from the previous studies on Kamoa (Kamoa 2013 PEA, Kamoa 2016 PFS, OreWin Kamoa-Kakula 2023 NI 43-101) undertaken in order to provide geotechnical designs for the room-and-pillar method incorporated in the Kamoa 2017 PFS.

Figure 16.11 Kakula Q-Contour map



Source: SRK, 2020

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 and smoothing effect. Ground support selection strategies are guided by the rock mass classification Q-system.

OHMS has recently completed a gap analysis and rock mass characterization (RMR, GSI), along with recommendations for future study work. These recommendations are outlined in the following sections.

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

Geological Strength Index (GSI)

The geological strength index (GSI) is another system of rock mass characterization generally determined visually from the Hoek GSI chart based on the rock mass degree of blockiness and the conditions of the discontinuity planes. GSI can also be estimated using the equation below, which contains certain Barton Q parameters:

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

- RQD is the rock quality designation
- Jr is the joint roughness
- Ja is the joint alteration

“The geological character of rock material, together with the visual assessment of the mass it forms, is used as a direct input to the selection of parameters relevant for the prediction of rock-mass strength and deformability…” (Marinos, et al., 2005).

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

Field stress measurements should be carried out during implementation of the mine projects to confirm the extensional normal faulting regime and the north-northeast approximate orientation of the maximum horizontal compressional stress. Available stress field data from Kamoto (KCC), some 30 km away from Kakula mine is listed in Table 16.3.

Table 16.3 Stress orientation and magnitude (Beck, 2025)

Stress magnitude	Trend	Plunge	Stress magnitude below surface (MPa)		
			500 m	600 m	650 m
σ1 = 0.027 * d	-	90°	13.5	16.2	17.6
σ2 = σ1	270°	0°	13.5	16.2	17.6
σ3 = σ1* 0.75	180°	0°	10.1	12.2	13.2

Laboratory tests undertaken or recommended to develop material properties include:

- Density
- UCS - Uniaxial compressive strength (MPa)
- UCM - Uniaxial compressive strength (MPa), Young’s Modulus and Poisson’s ratio
- UTB – Uniaxial Indirect tensile strength test (Brazilian method)
- TCS – Triaxial Compressive Strength (MPa)

The location of laboratory tests for each mine should be mapped to ensure that the appropriate material property parameters are used to assess specific locations with similar geotechnical characteristics. For example, given the size of the operations, blanket use of medium values will not be appropriate across each geotechnical domain for each rock type.

Based on the orebody orientation and proposed mining layouts, geotechnical assessments have considered the proposed mining methods that include:

- Room (Bord) and Pillar – shallow areas with ore dipping less than 15°.
- Low-angle long hole stoping (LHS) with cemented past fill backfill.
- Drift and Fill.

Mining methods are applied according to deposit thickness, dip gradient, structural complexity and depth. Flat, regular gradient areas use room and pillar methods, more complex areas apply drift and fill or stoping mining. Mining methods are described in more detail in section 16.4.3.

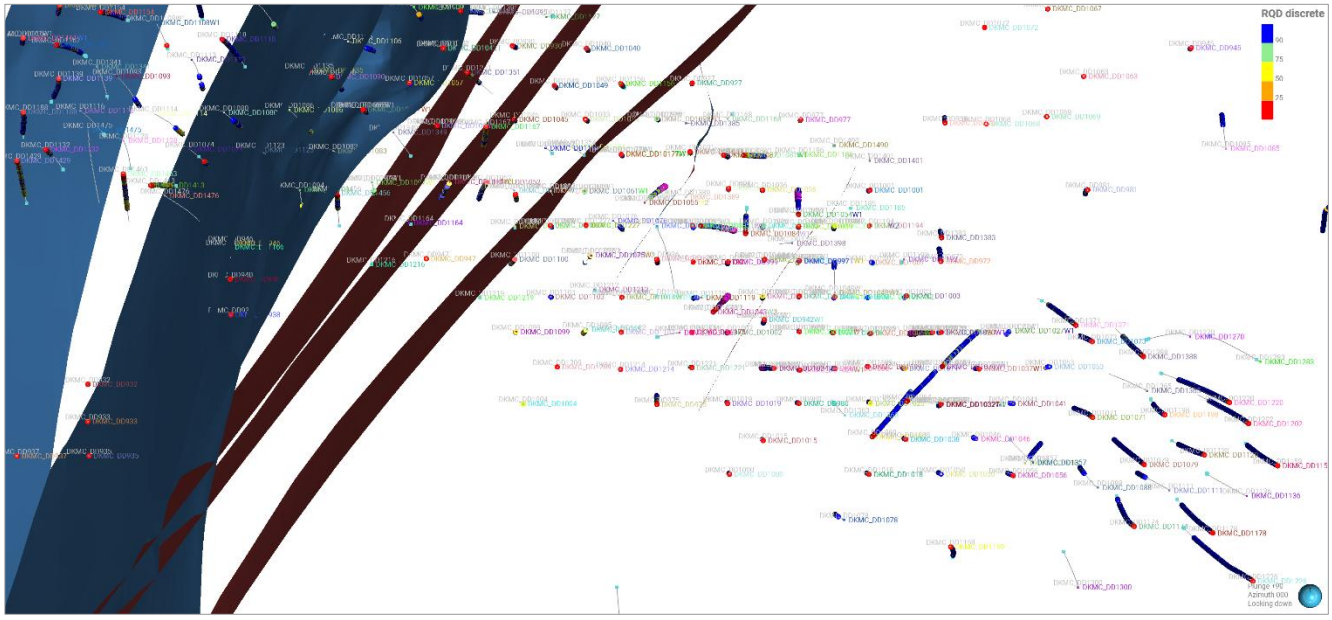
AMC concurs with Beck, 2025 for all relevant potential mining methods at Kamoa-Kakula, including drift and fill and LHS:

- A high transient void volume and a high extraction ratio will be present during intermediate excavation stages.
- Mining front lengths are a major factor affecting stress in work areas.
- Backfill limits the propagation of collapses in spans after they occur to some extent and is critical for mitigating air blast risk. There is also some benefit in reducing damage to the skin of future secondary stopes; however, pillar core stress and strength, which are the main factors in pillar capacity, are barely affected by the backfill in this environment.
- Stoping, filling, and development must all proceed together through milestones designed to manage hazards. These controls must be integral to the method.

16.2.5.2 Kakula and Kakula East geotechnical data summary

Geotechnical drillhole density across the Kakula mining area is indicated to have wide spaced even coverage in Figure 16.12.

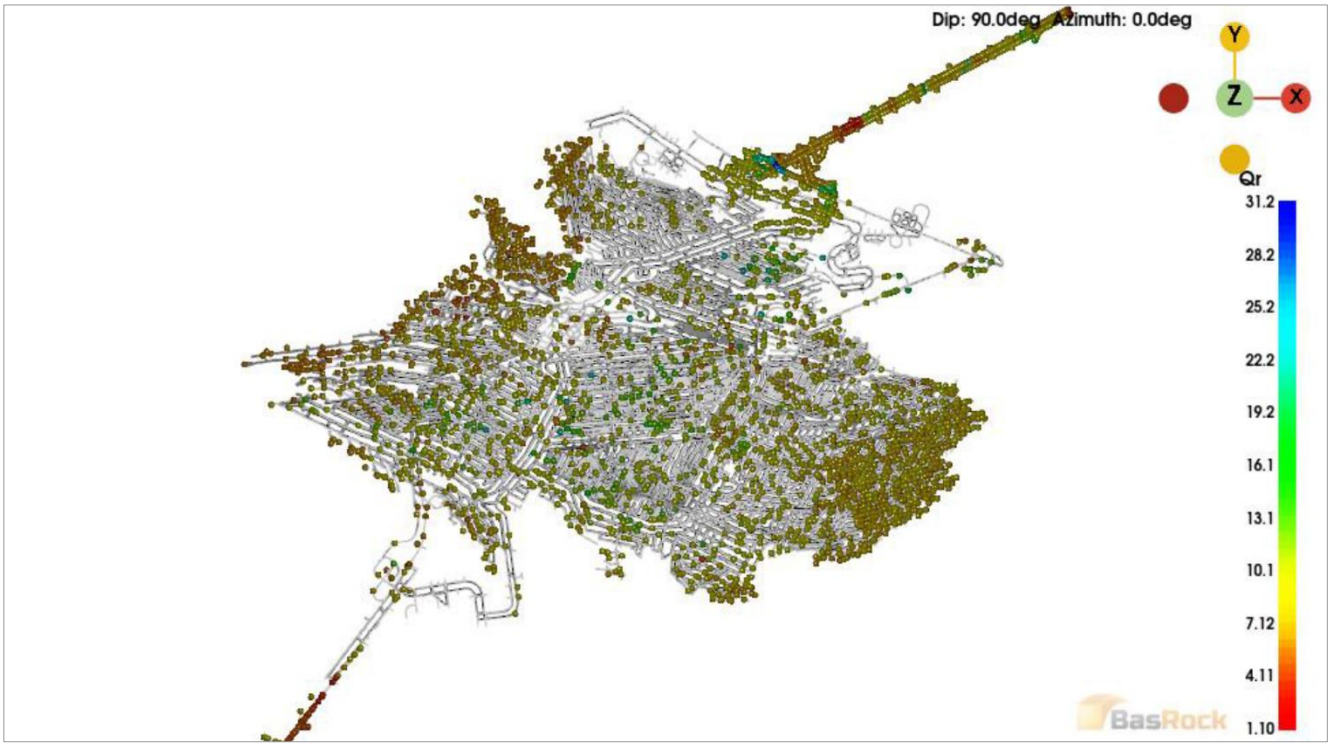
Figure 16.12 Schematic - Kakula geotechnical drillhole plan with faults



Source: KCSA 2025

Development mapping across Kakula sees a geotechnical database containing some 3,500 data points with x, y, z coordinates to enable spatial analysis. Q system analyses enabled the development of rock mass quality estimates across the Kakula mine as shown in Figure 16.13.

Figure 16.13 Kakula Q point estimate



Source: OHMS, 2025

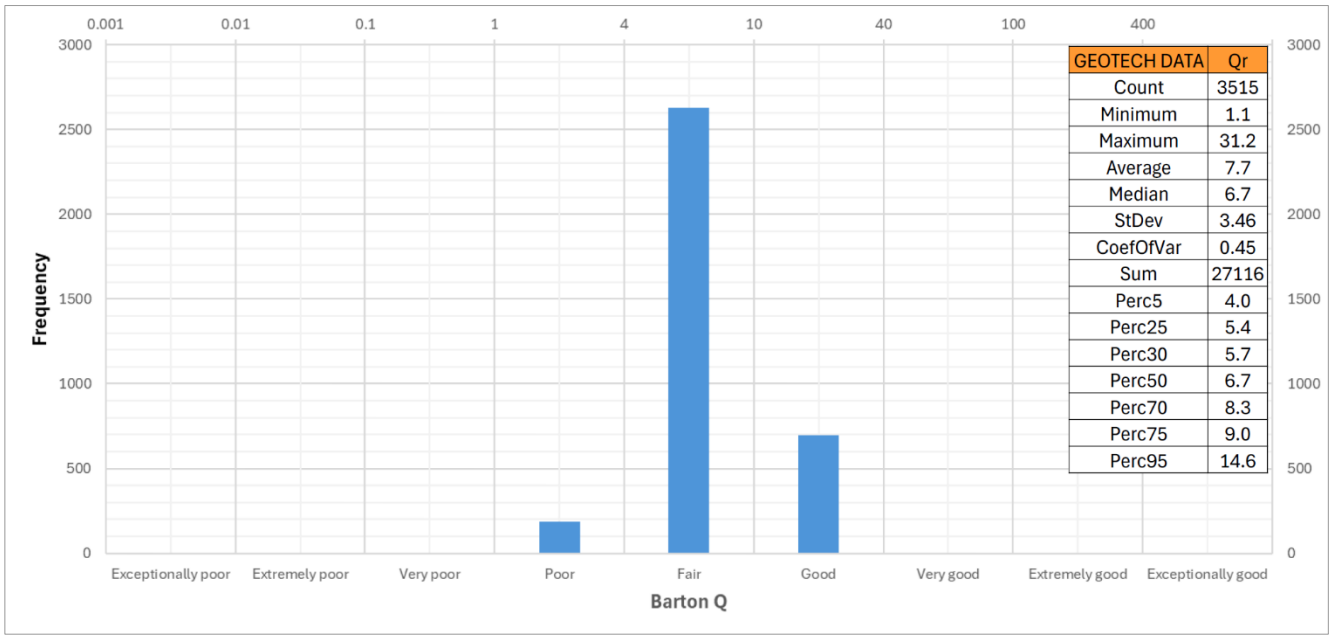
Figure 16.13 illustrates the maximum Q value is 31.2 and the minimum is 1.1. The average value is 7.7 with a standard deviation of 3.64 (OHMS, 2025). The Q system rates the ground conditions between poor and good.

It is not clear in the Q rock mass classification data displayed which of the three main rock types in the stratigraphic sequence are described:

- Diamictite (SDT)
- Siltstone (SSL)
- Sandstone (SST)

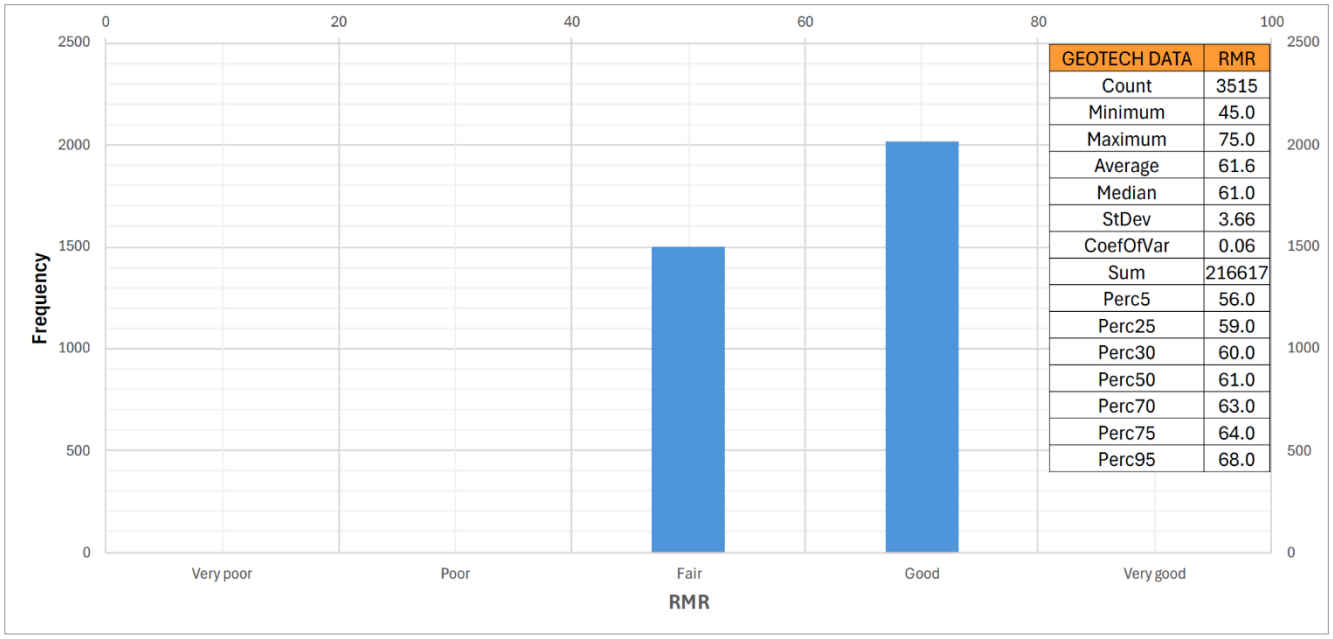
Details on the general location, such as backs (roof) or walls, and whether the poorer rock mass breccia unit has been included in the assessment is not evident. General rock mass conditions are shown in the frequency distributions in Figure 16.14 for Q rating and Figure 16.15 for RMR, fair (41-50) to good (61-80) rock mass.

Figure 16.14 Kakula frequency distribution for Q rating



Source: OHMS, 2025

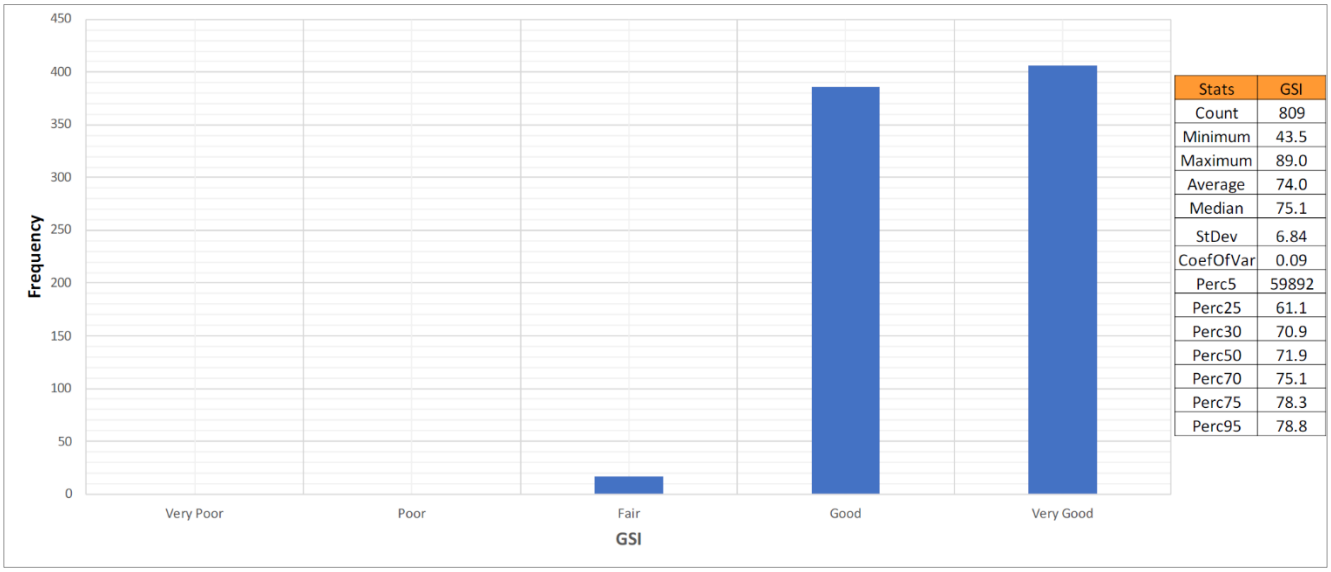
Figure 16.15 Kakula frequency distribution for RMR (OHMS, 2025)



Source: OHMS, 2025

Geological strength index (GSI) estimates the degree of rock mass blockiness and conditions of the discontinuity planes to be generally good to very good (Figure 16.16).

Figure 16.16 Kakula frequency distribution for GSI (OHMS, 2025)



Source: OHMS, 2025

A preliminary understanding of the rock mass quality is provided by the empirical rock mass assessments. Ongoing data collection and validation is required as mining develops to ensure the complex geological conditions encountered are considered with detailed modelling of mining methods and schedules.

Kakula laboratory testing material properties from the three main rock types are based on the following laboratory testing and summarized in Table 16.4:

- Density (kg/m^3)
- Uniaxial Compressive Strength (MPa)
- Young's Modulus (GPa)
- Poisson's Ratio

Intact rock properties from laboratory test samples have been used in quarterly modelling of Kakula by OHMS (Table 16.4).

Table 16.4 Kakula laboratory testing summary

Material property	SDT	SSL	SST
Density (kg/m^3)	2814	2945	2628
UCS (MPa)	122	139	227
Young's modulus (GPa)	66.081	55.471	72.875
Poisson's ratio	0.29	0.33	0.23

Source: OHMS, 2021
Note: Number of tests not specified

Post failure studies have identified a significant issue with the variability and uncertainty in uniaxial compressive strength (UCS) values across different reports. For example, Villaescusa (2025), reports that the Diamictite UCS values ranged from 70 MPa to over 120 MPa, while siltstone (SSL) was initially classified as weak (8–36 MPa) but later reported as exceeding 100 MPa in some studies. An upgrade of the

geotechnical database is required to ensure adequate sampling in different horizons, and validation as new data became available. The presence of weak horizons such as the breccia depicted in Figure 16.5, was not fully accounted for in previous hangingwall and pillar design. As experienced, these layers can reduce pillar strength and increase susceptibility to weathering.

To calibrate the Kakula failure in numerical modelling the material properties were revised to achieve a close match and comparison between modelled and measure surface movements. The revised material properties are listed in Table 16.5.

Table 16.5 Kakula revised material properties from numerical modelling

Rock types	WEATH	SDT	KPS	SSL	BRECCIA	SST
Density (kg/m^3)	2800	2800	2800	2800	2800	2800
UCS (MPa)	15	89	45	45	30	121
GSI	40	58	45	45	40	45
Young's modulus (GPa)*	5.35	24.57	9.76	11.71	6.99	25.00
Poisson's ratio	0.25	0.25	0.25	0.25	0.25	0.25

Source: Adapted Beck, 2025
Note: *Peak

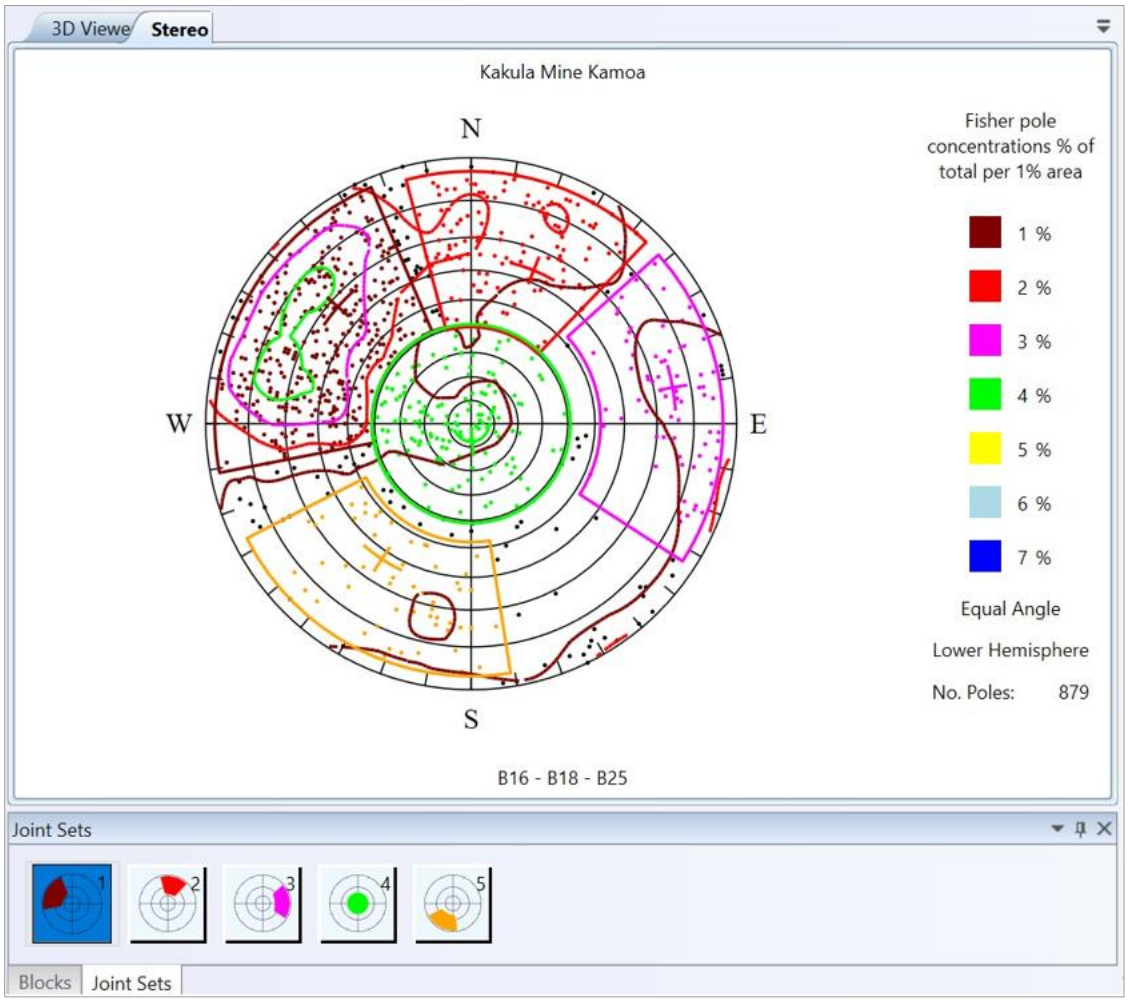
Kakula structural logging is displaying on the stereonet in Figure 16.17 and summarized below:

- Data available is predominantly from sub-vertical drillholes (879 data points).
- Five structures sets including a dominant flat dipping structure set (bedding).
- The orientation of structure sets form gravity falling and sliding wedge failure modes.

Structural logging and mapping should be undertaken as the underground mine is re-opened to confirm ground support requirements and validate rock mass parameters.

Ground support designs are based on Q ratings. Where there are changes to the mining method or profile size, additional ground support is required to address the risk of structurally controlled failures, especially in areas where spans are increased.

Figure 16.17 Kakula main structure sets from SAFEX software



Source: Villaescusa, 2025

Preliminary understanding for mine designs must be validated on development to ensure that the complex geological conditions are well understood. Rock mass variability should be accounted for in both global stability and local design assessments. For example, pillar design must focus on both individual and surrounding pillars to ensure the potential for damage in the stoping blocks or areas is identified.

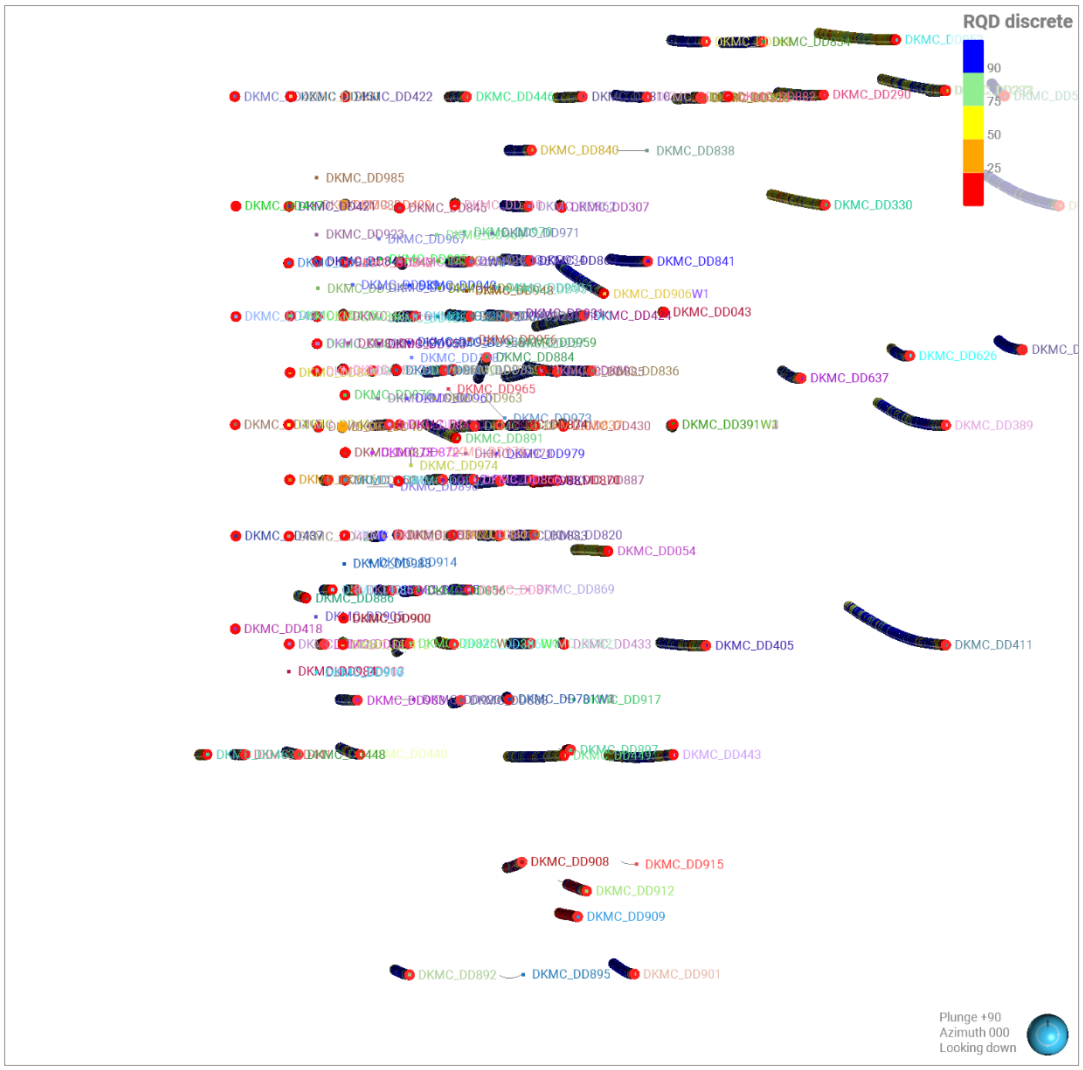
On going rock mass characterization during development and mining for design validation must include:

- Strength testing.
- Joint set characterization.
- In situ stress measurement.
- Understanding of potential for slip on pre-existing discontinuities.

16.2.5.3 Kansoko geotechnical data summary

Geotechnical drillhole density across the Kansoko mining area is indicated to be wide spaced with an even coverage as shown in Figure 16.18.

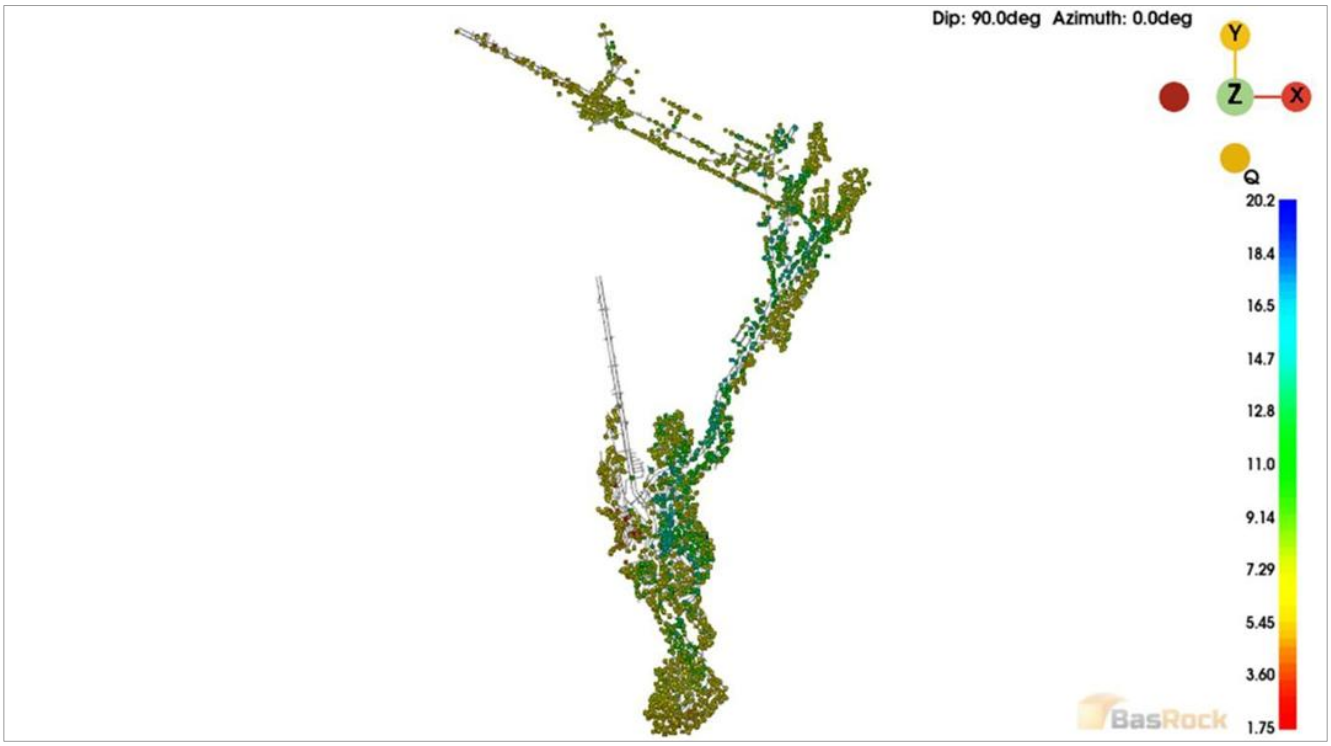
Figure 16.18 Schematic - Kansoko geotechnical drillhole plan



Source: KCSA 2025

Development mapping across Kansoko provides a geotechnical database containing nearly 3,000 data points with x, y, z coordinates to enable spatial analysis. Q system analyses enabled the development of rock mass quality estimates across the Kansoko mine as shown in Figure 16.19 based on quarterly numerical modelling assessment reporting by OHMS, 2025.

Figure 16.19 Schematic - Kansoko Q point estimate



Source: OHMS, 2025

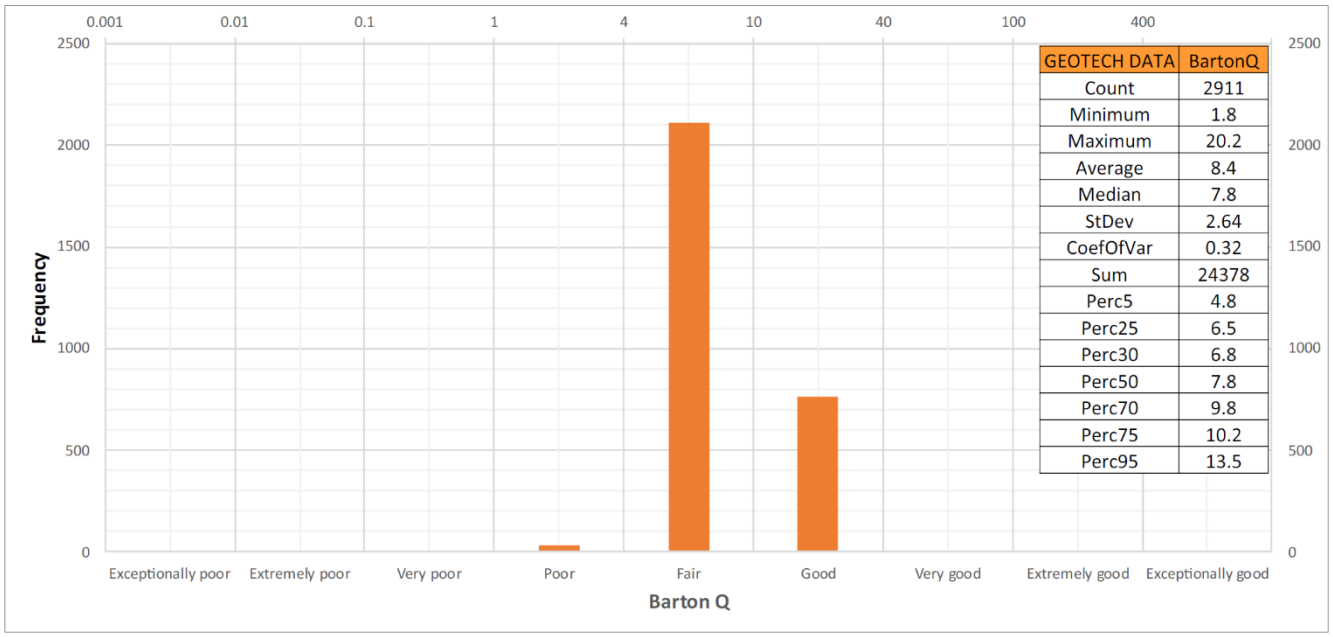
Figure 16.19 illustrates the maximum Q value is 20.2 and the minimum is 1.75. The average value is 8.6 with a standard deviation of 8.4 (OHMS, 2025). The Q system rates the ground conditions between poor and good.

Q rock mass classification data displayed which of the three main rock types in the stratigraphic sequence are described:

- Diamictite (SDT)
- Siltstone (SSL)
- Sandstone (SST)

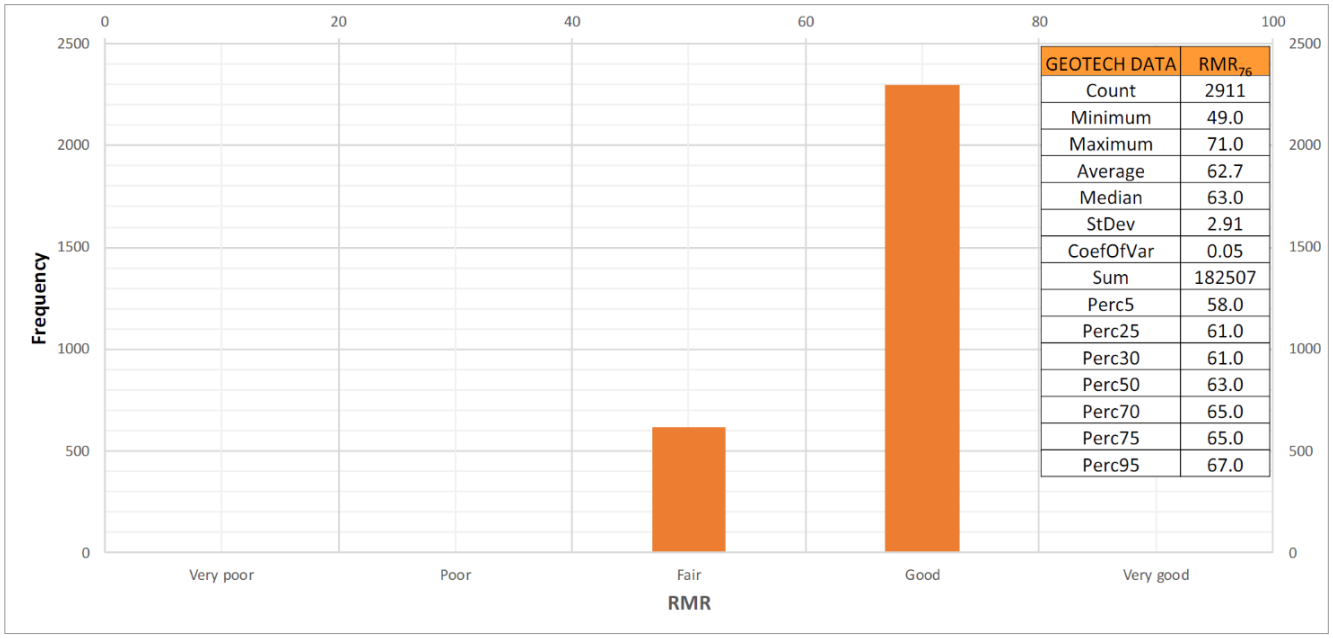
Details on the general location, such as backs (roof) or walls, and whether the poorer rock mass breccia unit, Kamoa Pyritic Siltstone (KPS), has included in the assessment is not evident and should be considered in future modelling. General rock mass conditions are shown in the frequency distributions in Figure 16.20 for Q rating and Figure 16.21 for RMR in the range of 49 to 71 classified as fair to good rock mass.

Figure 16.20 Kansoko frequency distribution for Q rating



Source: OHMS, 2025

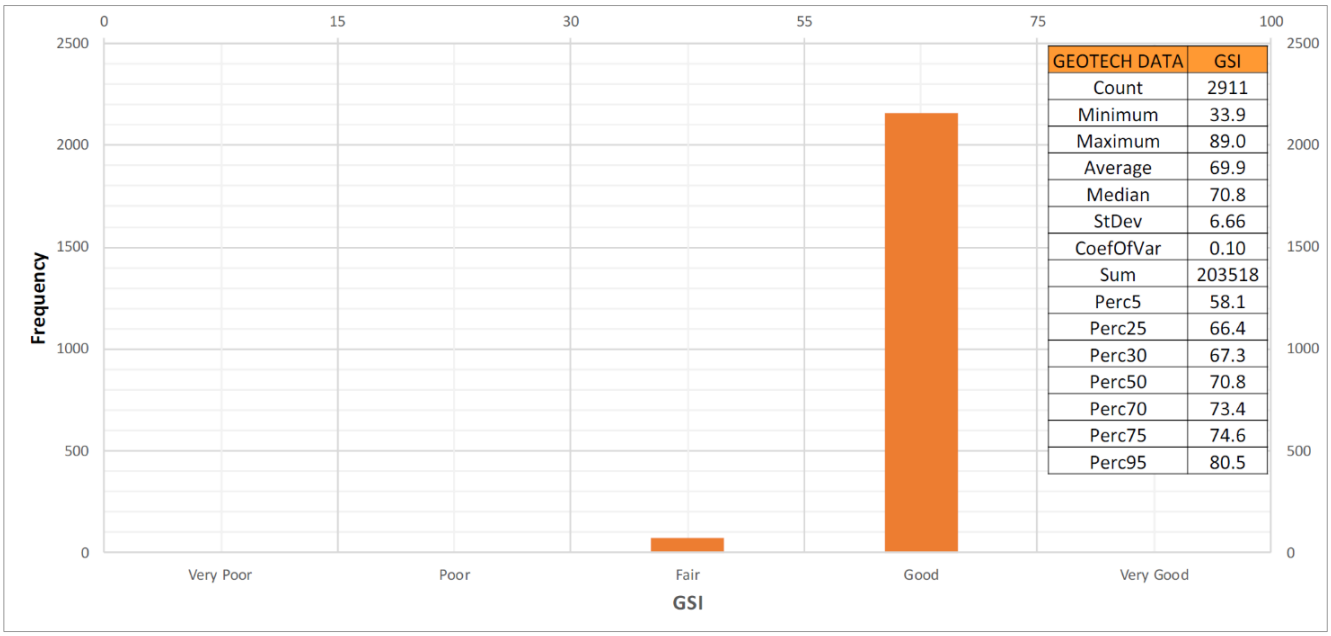
Figure 16.21 Kansoko frequency distribution for RMR



Source: OHMS, 2025

Geological strength index (GSI) estimates the degree of rock mass blockiness and conditions of the discontinuity planes to be generally good to very good (Figure 16.22).

Figure 16.22 Kansoko frequency distribution for GSI



Source: OHMS, 2025

A preliminary understanding of the rock mass quality is provided by the empirical rock mass assessments. Ongoing data collection and validation is required as mining develops to ensure the complex geological conditions encountered are considered with detailed modelling of mining methods and schedules.

Kansoko laboratory testing material properties from the three main rock types are based on the following laboratory testing and summarized in Table 16.6:

- Density (kg/m^3)
- Uniaxial Compressive Strength (MPa)
- Young's Modulus (GPa)
- Poisson's Ratio

Laboratory test samples are based on a reasonable spread of drillholes across the mine footprint:

- Only UCS and UCM testing were conducted on the selected samples inside the Kansoko mining area.
- Based on the TLDC assessment more testing is needed.
- Recommendations to perform TCS and UTB tests to gain insight into the effects of confinement on both the rock's strength and its tensile properties have been provided.

Table 16.6 Kansoko laboratory testing summary

Material property	Rock unit	SDT	SSL	SST
Density (kg/m^3)	Number of tests	-	-	-
	Mean	**2253**	**2662**	**2603**
UCS (MPa)	Number of tests	20	10	6
	Minimum	81	131	91
	Mean	**89**	**167**	**138**
	Maximum	98	202	186
	Standard deviation	18	49	45
Young's modulus (GPa)	Number of tests	12	10	4
	Minimum	49.266	47.910	23.948
	Mean	**63.054**	**58.657**	**53.738**
	Maximum	77.800	85.5	76.600
	Standard deviation	13.723	15.639	19.234
Poisson's ratio	Number of tests	28	7	8
	Minimum	0.12	0.16	0.18
	Mean	**0.27**	**0.30**	**0.24**
	Maximum	0.34	0.44	0.29
	Standard deviation	0.05	0.09	0.04

Source: OHMS, 2025

Elastic material properties (Hoek and Diederichs, 2006) listed in Table 16.7.

Table 16.7 Kansoko elastic material properties (OHMS, 2025)

Material property	Young's modulus (GPa)	Poisson's ratio	Shear modulus (GPa)	Normal modulus (GPa)
Host rock	24.142	0.25	22.076	36.793

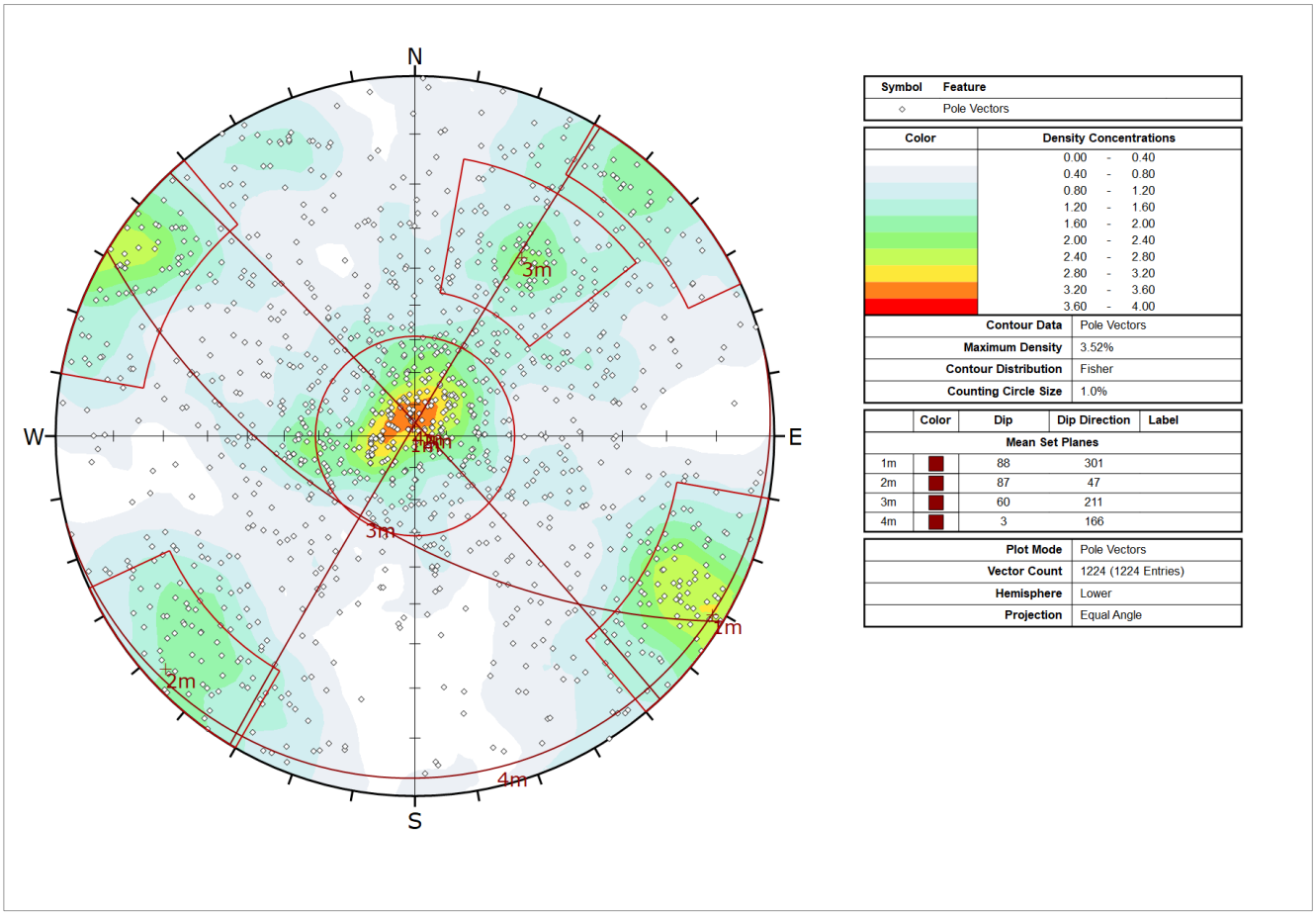
Kansoko structural logging information is well spread across the mining layout.

- Data available is predominantly from steep angled drillholes.
- A subset of structural data (1224 points) indicates four structures sets including a dominant flat dipping structure set (bedding) as shown in Figure 16.23.
- The orientation of structure sets form gravity falling and sliding wedge failure modes.

Structural logging and mapping should be ongoing to confirm ground support requirements and validate rock mass parameters and mine design elements.

Ground support designs are based on Q ratings. Where there are changes to the mining method or profile size, additional ground support is required to address the risk of structurally controlled failures, especially in areas where spans are increased.

Figure 16.23 Kansoko main structure sets



Source: KCSA 2025

Preliminary rock mass understanding and mine designs must be validated on development to ensure that the complex geological conditions are well understood. Rock mass variability should be accounted for in both global stability and local design assessments.

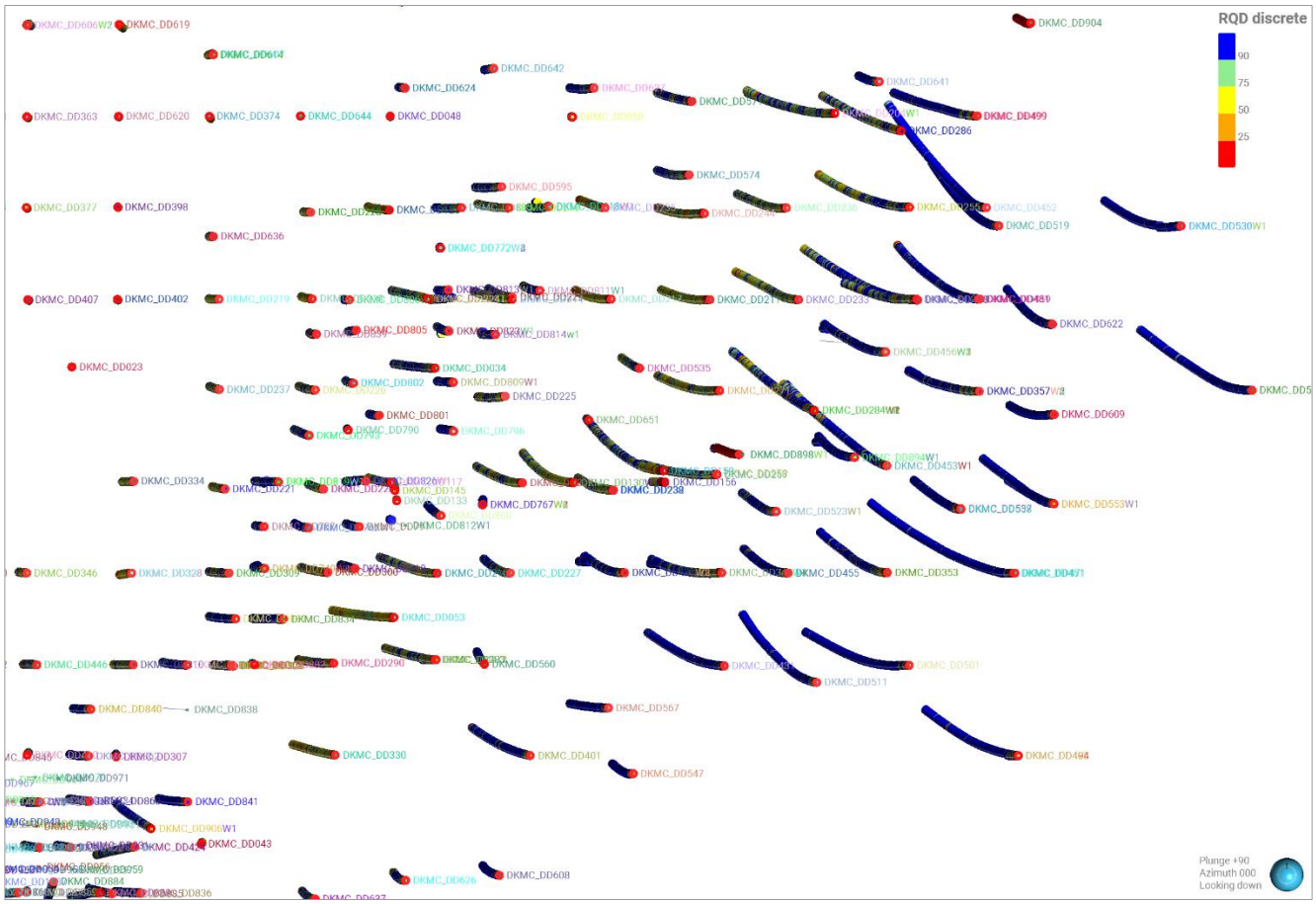
On going rock mass characterization during development and mining must include:

- Strength testing.
- Joint set characterization.
- In situ stress measurement.
- Understanding of potential for slip on pre-existing discontinuities.

16.2.5.4 Kamoa 1 geotechnical data summary

Geotechnical drillhole density across the Kamoa 1 mining area is indicated to be wide spaced with a fairly even coverage as shown in Figure 16.24. Drill spacing is between 200 and 400 m with infill drilling planned to 80 m.

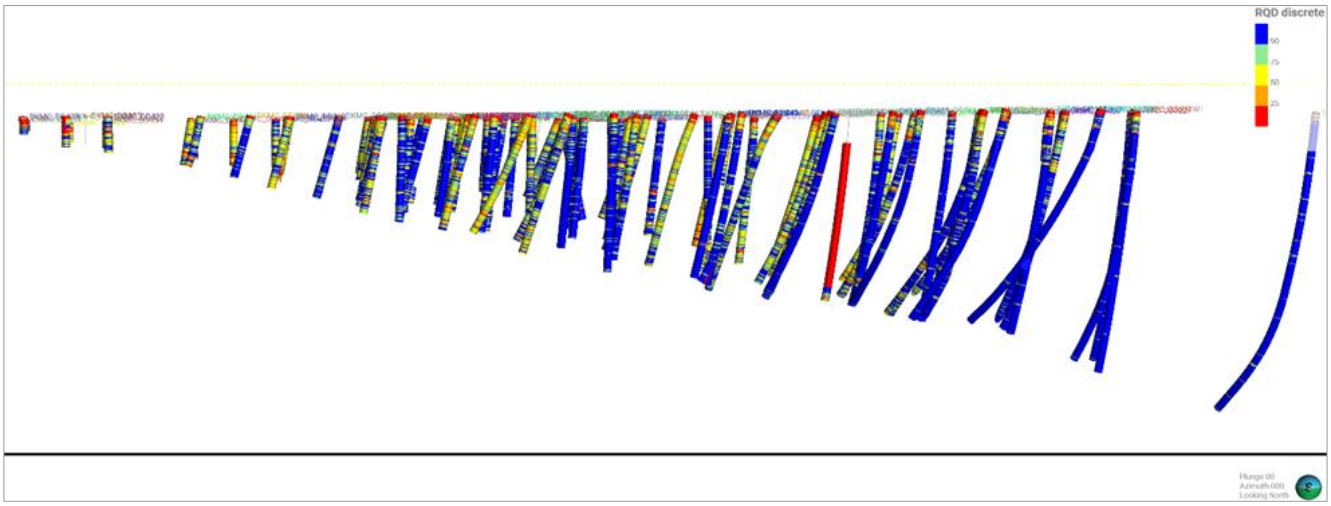
Figure 16.24 Schematic - Kamoa 1 geotechnical drillhole plan with RQD



Source: KCSA 2025

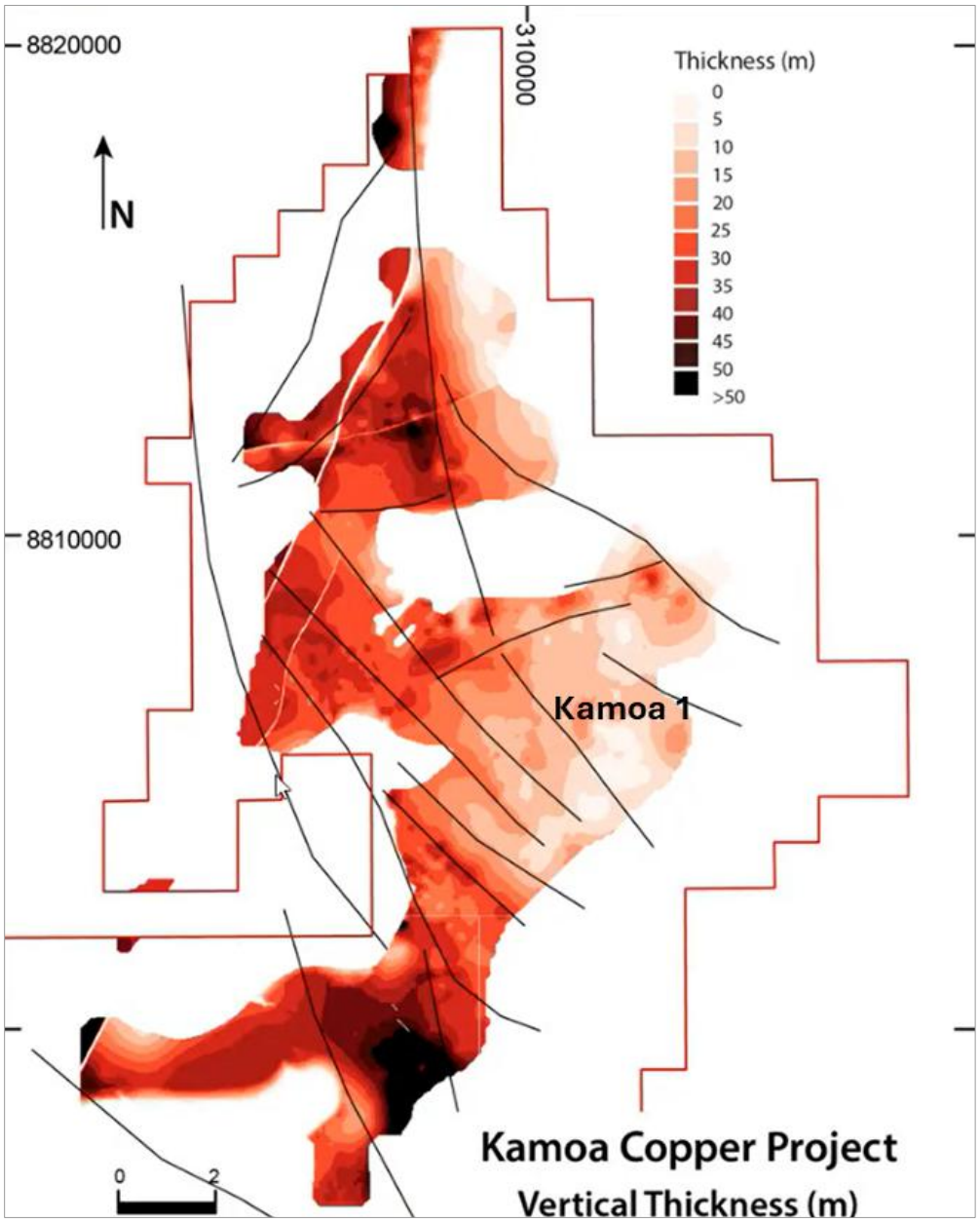
A cross section through the central area showing RQD provides an overview of the flat to moderately dipping orebody and generally fair (yellow) to good (blue) rock mass conditions (Figure 16.25). Primary depositional faults across Kamoa are described as not being associated with poor ground (Figure 16.26) and are oriented to the northwest.

Figure 16.25 Kamoa 1 cross section showing RQD looking north



Source: KCSA 2025
Note: 'Red' drillhole not confirmed to be very poor ground or fault related.

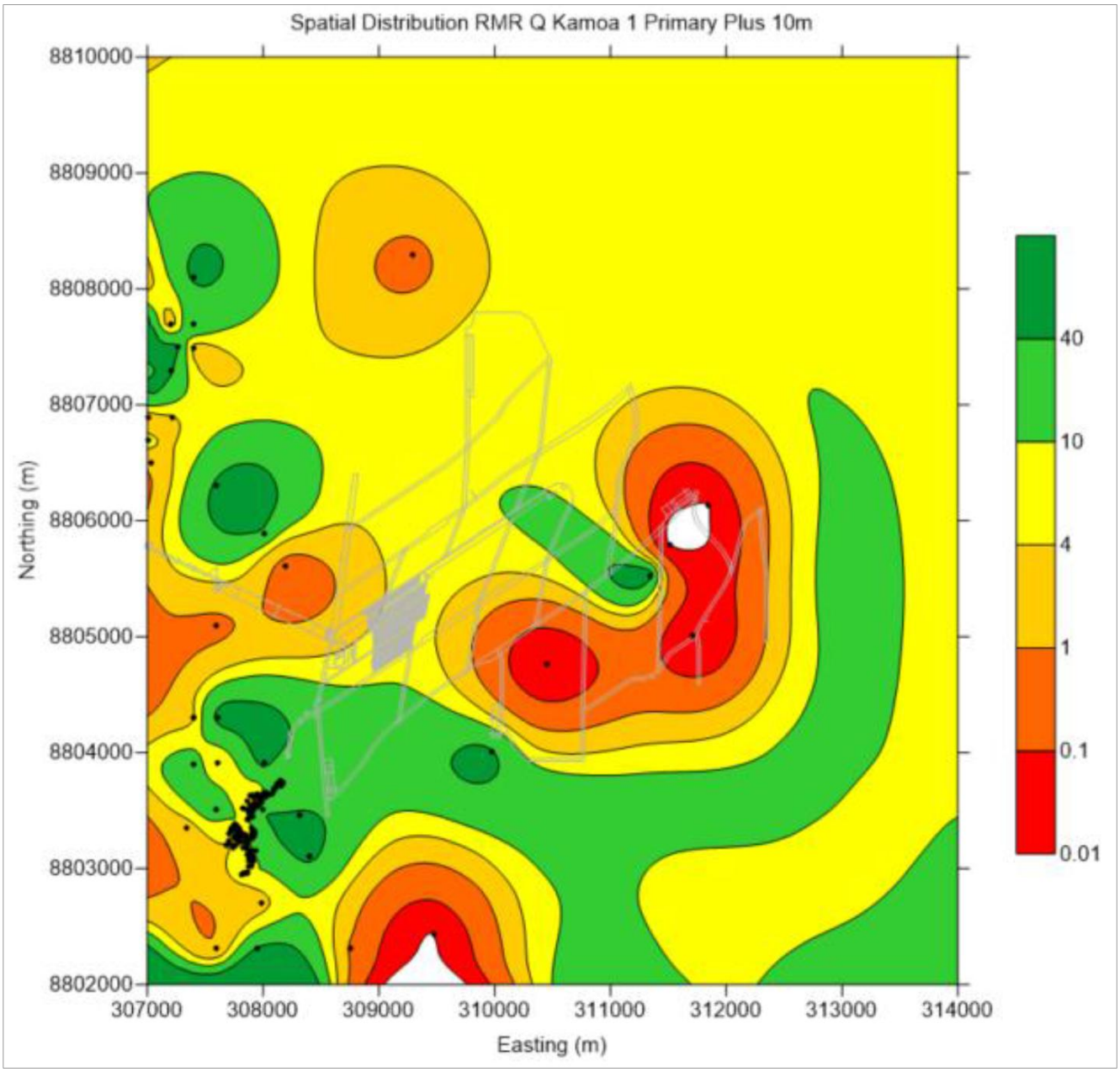
Figure 16.26 Kamoa 1 primary depositional faults plan



Source: KCSA 2025

Ongoing data collection is required to progress the Kamoa 1 geotechnical database from the low confidence spatial variability in the interpolated data SRK Q-contours (Figure 16.27).

Figure 16.27 Kamoa 1 Q-contours SRK estimate



Source: OreWin, 2023

Geotechnical domains are classified by the three main rock types in the stratigraphic sequence are described:

- Diamictite (SDT)
- Siltstone (SSL)
- Sandstone (SST)

Intact rock properties specific to Kamoa 1 are not defined in recent reporting. The Kamoa-Kansoko life-of-mine numerical modelling study (OHMS, 2024) has combined all available laboratory test results from

2022, 2023, and 2024. Details on the general location, such as backs (roof) or walls, and whether the poorer rock mass breccia unit, Kamoa Pyritic Siltstone (KPS), has not been included and should be considered in future modelling.

Ongoing data collection and validation is required during mining to ensure the complex geological conditions encountered are considered with detailed modelling of mining methods and schedules.

Generalized laboratory testing material properties for the three main rock types are based on the following laboratory testing and summarized in Table 16.8 to Table 16.10:

- Uniaxial Compressive Strength (MPa)
- Young's Modulus (GPa)
- Poisson's Ratio

Laboratory test results were classified to identify outliers and to statistical assessment to quantify the variability using all available data. Further data should be collected to:

- Test UCS and UCM specific to the Kamoa 1 mining area.
- Perform TCS and UTB tests to gain insight into the effects of confinement on both the rock's strength and its tensile properties.

AMC considers that each mine area should have its own laboratory test data set to ensure variability in the rock mass is considered. A thorough review of using the generalized data should be undertaken prior to design assessments and numerical modelling.

Table 16.8 Kamoa-Kansoko laboratory testing summary – SDT

Material property		**SDT**	
UCS (MPa)	**Current**	**Combined**	**Combined filtered**
Number of tests	30	48	28
Minimum	39	45	50
Mean	47	53	58
Standard deviation	23	27	19
Maximum	56	61	65
Young's modulus GPa	**Current**	**Combined**	**Combined filtered**
Number of tests	30	48	28
Minimum	22.373	29.987	31.379
Mean	32.130	36.913	37.657
Standard deviation	26.131	23.850	16.190
Maximum	41.888	43.838	43.935
Poisson's ratio	**Current**	**Combined**	**Combined Filtered**
Number of tests	30	48	28
Minimum	0.22	0.23	0.22
Mean	0.23	0.24	0.24
Standard deviation	0.04	0.06	0.06
Maximum	**0.25**	**0.26**	**0.26**

Source: OHMS, 2025
Note: Values in bold used in OHMS, 2025 assessments

Table 16.9 Kamoa-Kansoko laboratory testing summary – SSL

Material property	**SSL**			
UCS (MPa)	**Unfiltered**	**Filtered**	**Historic**	**Historic filtered**
Number of tests	17	11	12	10
Minimum	26	33	110	131
Mean	35	43	149	167
Standard deviation	18	15	61	49
Maximum	44	53	188	202
Young's modulus GPa	**Unfiltered**	**Filtered**	**Historic**	**Historic filtered**
Number of tests	17	11	12	10
Minimum	12.068	16.485	43.486	47.910
Mean	20.320	27.452	50.825	54.330
Standard deviation	16.051	16.325	11.551	8.974
Maximum	28.573	38.419	58.164	60.750
Poisson's ratio	**Unfiltered**	**Filtered**	**Historic**	**Historic filtered**
Number of tests	17	11	12	10
Minimum	0.23	0.20	0.23	0.23
Mean	0.26	0.23	0.31	0.32
Standard deviation	0.05	0.05	0.11	0.12
Maximum	**0.29**	**0.27**	**0.38**	**0.41**

Source: OHMS, 2025
Note: Values in bold used in OHMS, 2025 assessments

Table 16.10 Kamoa-Kansoko laboratory testing summary – SST

Material property	**SST**			
UCS (MPa)	**Current**	**Combined**	**Filtered hardness**	**Filtered strength**
Number of tests	24	37	18	24
Minimum	26	30	28	30
Mean	32	42	35	35
Standard deviation	13	35	14	13
Maximum	37	54	41	41
Young's modulus GPa	**Current**	**Combined**	**Filtered hardness**	**Filtered strength**
Number of tests	24	37	18	24
Minimum	18.206	18.922	15.961	14.808
Mean	24.170	24.824	21.779	19.526
Standard deviation	14.124	17.702	11.699	11.172
Maximum	30.134	30.726	27.597	24.243
Poisson's ratio	**Current**	**Combined**	**Filtered hardness**	**Filtered strength**
Number of tests	24	37	18	24
Minimum	0.22	0.25	0.22	0.25
Mean	0.26	0.29	0.27	0.30
Standard deviation	0.08	0.12	0.08	0.11
Maximum	0.29	0.33	0.31	0.34

Source: OHMS, 2025
Note: Values in bold used in OHMS, 2025 assessments

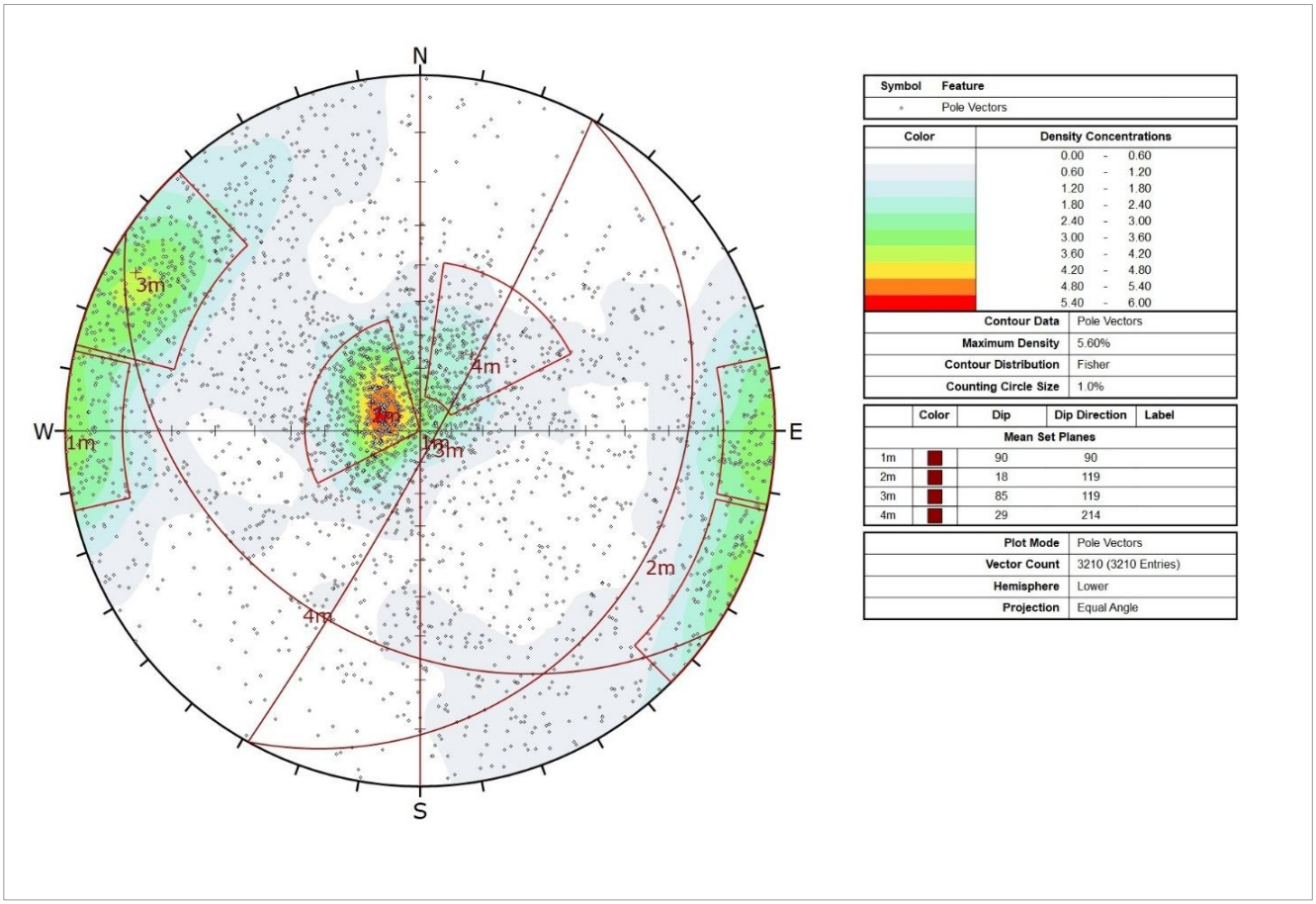
Kamoa 1 structural logging information is well spread across the mining layout:

- Data available is predominantly from steep angled drillholes.
- A subset of structural data (3210 points) indicates four structures sets including a dominant flat dipping structure set (bedding) as shown in Figure 16.28.
- The orientation of structure sets form gravity falling and sliding wedge failure modes.

Structural logging and mapping should be ongoing to confirm ground support requirements and validate rock mass parameters and mine design elements.

Ground support designs are based on Q ratings. Where there are changes to the mining method or profile size, additional ground support is required to address the risk of structurally controlled failures, especially in areas where spans are increased.

Figure 16.28 Kamoa 1 main structure sets



Source: KCSA 2023

Preliminary rock mass understanding and mine designs must be validated on development to ensure that the complex geological conditions are well understood. Rock mass variability should be accounted for in both global stability and local design assessments.

On going rock mass characterization during development and mining must include:

- Strength testing.
- Joint set characterization.
- In situ stress measurement.
- Understanding of potential for slip on pre-existing discontinuities.

16.2.5.5 Kamoa 2 geotechnical data summary

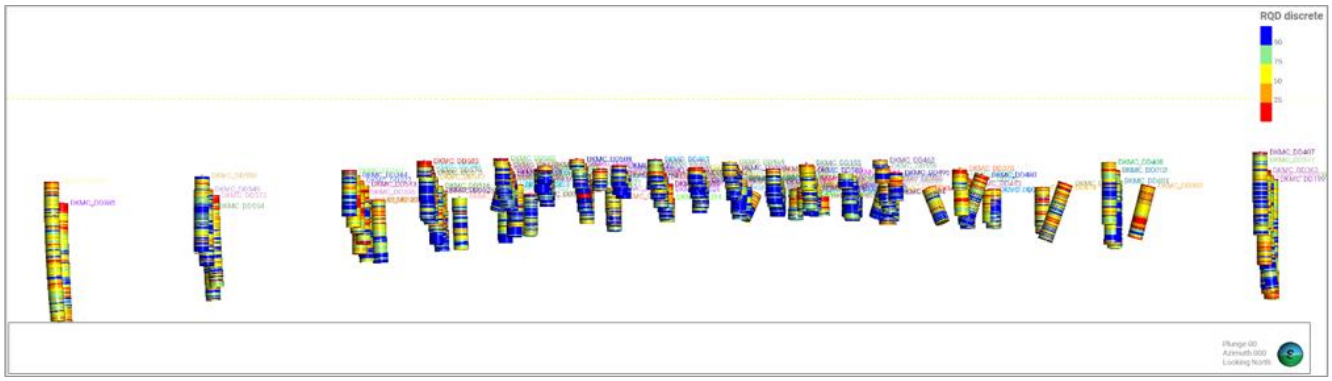
Geotechnical drillhole density across the Kamoa 2 mining area is indicated to be wide spaced with a fairly even coverage as shown in Figure 16.29. Drill spacing is between 200 and 400 m with infill drilling planned to 80 m.

Figure 16.29 Kamoa 2 geotechnical drillhole plan with RQD

Source: KCSA 2026

A cross section through the central area showing RQD provides an overview of the near surface relatively flat dipping orebody with variable rock mass conditions (Figure 16.30). Future assessments will also consider the viability of an open pit at Kamoa 2.

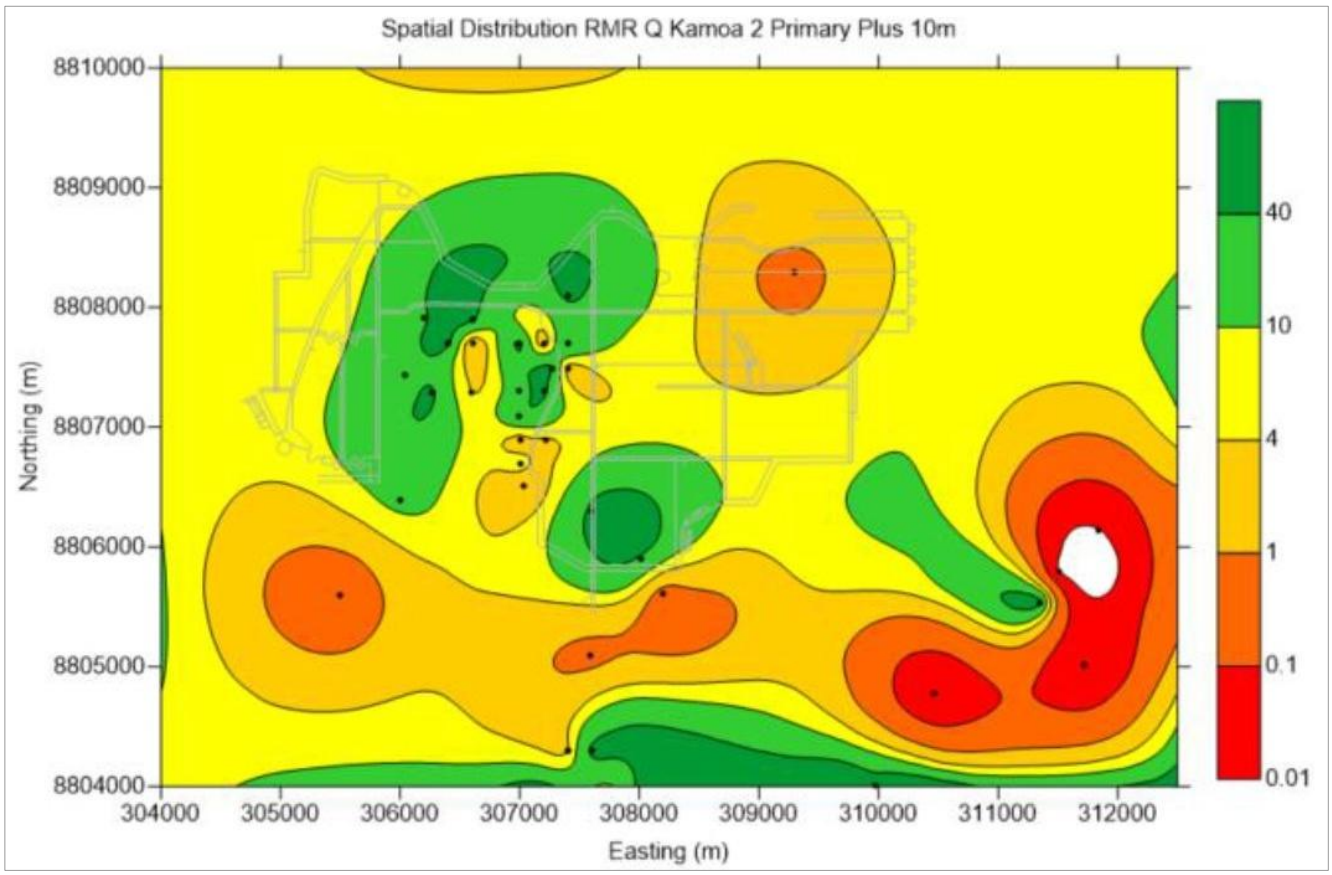
Figure 16.30 Kamoa 2 cross section showing RQD looking North



Source: KCSA 2025

Ongoing data collection is required to progress the Kamoa 2 geotechnical database from the low confidence spatial variability in the interpolated data SRK Q-contours (Figure 16.31).

Figure 16.31 Kamoa 2 Q-contours SRK estimate



Source: OreWin, 2023

Geotechnical domains are classified by the three main rock types in the stratigraphic sequence are described:

- Diamictite (SDT)
- Siltstone (SSL)
- Sandstone (SST)

Intact rock properties specific to Kamoa 2 are not defined in recent reporting. The Kamoa-Kansoko life-of-mine numerical modelling study (OHMS, 2024) has combined all available laboratory test results from 2022, 2023, and 2024. Details on the general location, such as backs (roof) or walls, and whether the poorer rock mass breccia unit, Kamoa Pyritic Siltstone (KPS), has not been included and should be considered in future modelling.

Ongoing data collection and validation is required during mining to ensure the complex geological conditions encountered are considered with detailed modelling of mining methods and schedules.

Generalized laboratory testing material properties for the three main rock types are based on the following laboratory testing and are summarized in the previous report section in Table 16.8 to Table 16.10.

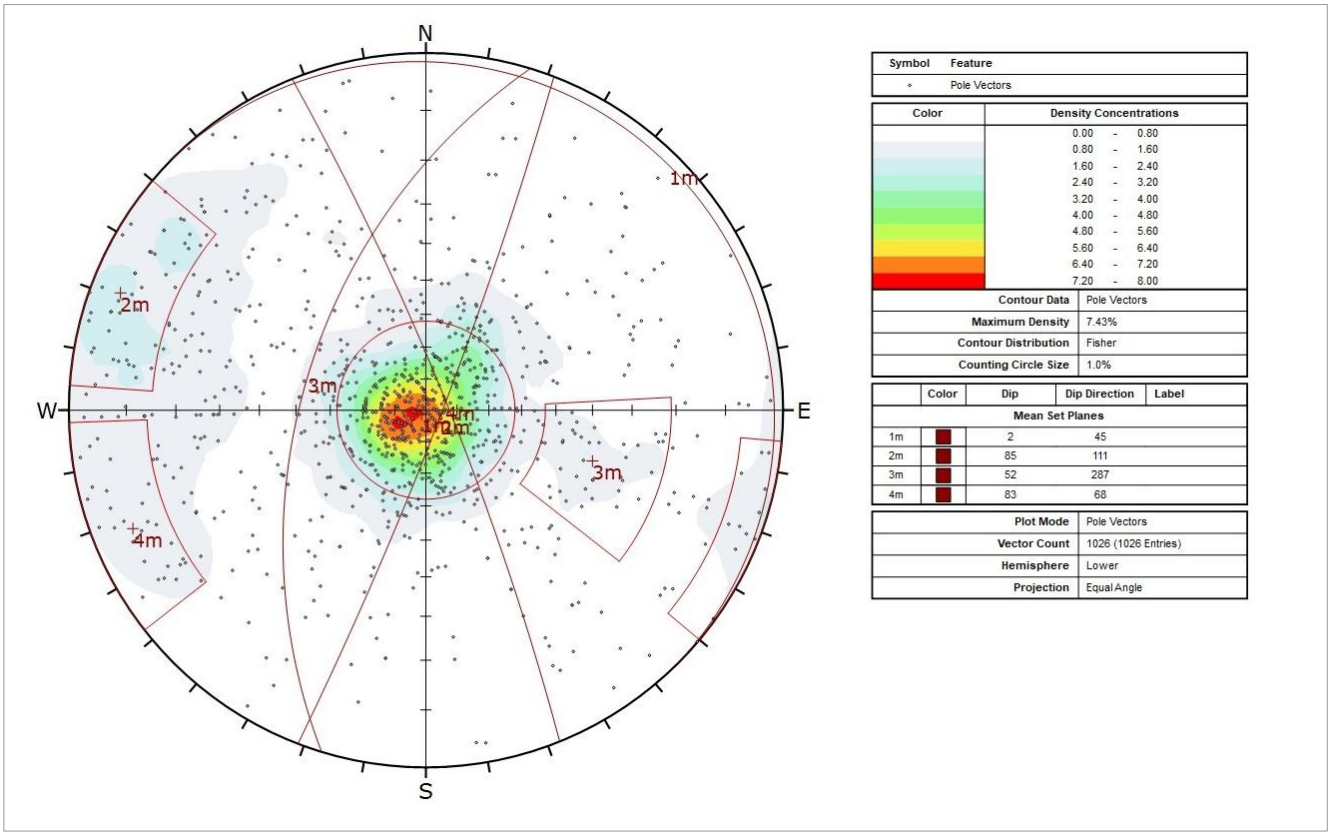
AMC considers that each mine area should have its own laboratory test data set to ensure variability in the rock mass is considered. A thorough review of using the generalized data should be undertaken prior to design assessments and numerical modelling.

Kamoa 2 structural logging is summarized on the stereonet in Figure 16.32:

- Limited structural logging information is available for Kamoa 2, approximately ten widely spaced drillholes.
- Data available is predominantly from angled drillholes.
- Structural data (1026 points) indicate two strong structures sets, likely four sets, including a dominant flat dipping structure set (bedding) as shown in Figure 16.32.
- The orientation of structure sets form mainly gravity falling failure modes.
- Structural logging is recommended as the underground mine is being developed.
- Ongoing structural logging will be required as the underground mine is being developed to confirm ground support requirements.
- Structural logging and mapping should be ongoing to confirm ground support requirements and validate rock mass parameters and mine design elements.

Ground support designs are based on Q ratings. Where there are changes to the mining method or profile size, additional ground support is required to address the risk of structurally controlled failures, especially in areas where spans are increased.

Figure 16.32 Kamoa 2 main structure sets

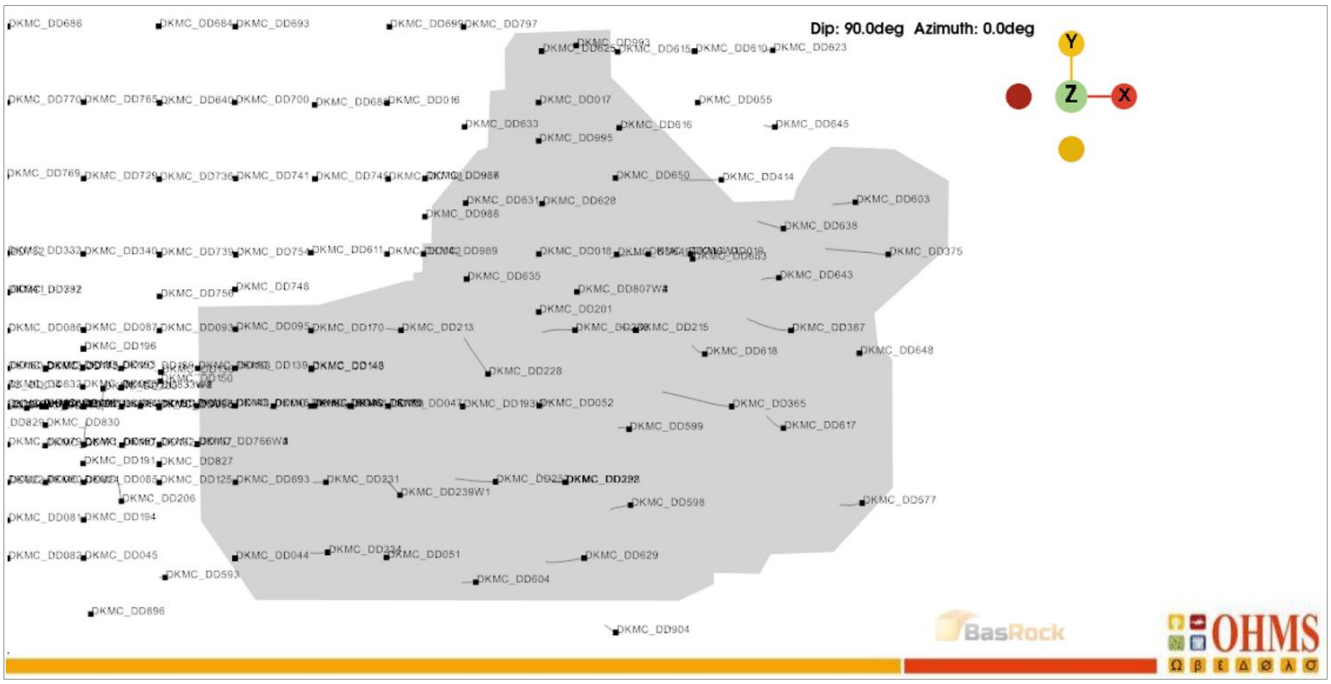


Source: KCSA 2023

16.2.5.6 Kamoa 3 geotechnical data summary

The geotechnical gap analysis (OHMS, 2025) of the Kamoa 3 area indicates that the density of the geotechnical drillholes are wide spaced with fairly even coverage as shown in Figure 16.33.

Figure 16.33 Kamoa 3 geotechnical drillhole plan



Source: OHMS, 2025

Based on the RMR obtained from geotechnical logging, the rock mass conditions for the three main rock types range as follows:

- Diamictite ‘poor’ to ‘good’ rock mass conditions (SDT)
- Siltstone ‘poor’ to ‘good’ rock mass conditions (SSL)
- Sandstone ‘very poor’ to ‘good’ rock mass conditions (SST)

Rock mass characterization data for Kamoa 3 is summarized in Table 16.11.

Table 16.11 Kamoa 3 rock mass characterization summary (adapted OHMS, 2025)

Material property	Statistical Distribution	RQD	RMR	GSI
SDT	25th percentile	90	57	52
	50th percentile	98	60	61
	75th percentile	100	67	65
	Average	**91**	**60**	**57**
SSL	25th percentile	80	53	48
	50th percentile	96	66	55
	75th percentile	100	67	62
	Average	**86**	**59**	**54**
SST	25th percentile	80	56	51
	50th percentile	94	63	58
	75th percentile	100	70	65
	Average	**86**	**61**	**56**

Kamoa 3 laboratory testing material properties from the three main rock types are based on the following laboratory testing and summarized in Table 16.12:

- Density (kg/m^3)
- Uniaxial Compressive Strength (MPa)
- Young's Modulus (GPa)
- Poisson's Ratio

Laboratory test samples are based on a reasonable spread of drillholes across the mine footprint:

- Only UCS and UCM testing were conducted on the selected samples inside the Kamoa 3 mining area.
- Based on the TLDC assessment more testing is needed.
- Recommendations to perform TCS and UTB tests to gain insight into the effects of confinement on both the rock's strength and its tensile properties have been provided.

Table 16.12 Kamoa 3 laboratory testing summary

Material Property	**Rock Unit**	**SDT**	**SSL**	**SST**
Density (kg/m^3)	Number of tests	31	7	9
	Minimum	2446	2937	2393
	Mean	**2780**	**3102**	**2593**
	Maximum	2995	3324	2785
	Standard deviation	127	128	159
UCS (MPa)	Number of tests	31	7	9
	Minimum	20	89	64
	Mean	**112**	**154**	**181**
	Maximum	225	201	288
	Standard deviation	51	37	89
Young's modulus (GPa)	Number of tests	28	7	8
	Minimum	30.900	44.100	30.500
	Mean	**63.054**	**58.657**	**53.738**
	Maximum	77.800	85.5	76.600
	Standard deviation	13.723	15.639	19.234
Poisson's ratio	Number of tests	28	7	8
	Minimum	0.12	0.16	0.18
	Mean	**0.27**	**0.30**	**0.24**
	Maximum	0.34	0.44	0.29
	Standard deviation	0.05	0.09	0.04

Additional laboratory tests recommended by OHMS, 2025 are listed in the Table 16.13.

Table 16.13 Kamoa 3 laboratory testing recommendations for further work

Rock types	Test Type	Number of Current Tests	Number of Additional Tests
SDT	TCS	0	24
	UCM	28	0
	UTB	0	16
SSL	TCS	0	24
	UCM	7	1
	UTB	0	16
SST	TCS	0	24
	UCM	8	0
	UTB	0	16

Kamoa 3 structural logging is listed in Table 16.14 and summarized below:

- The density of the structural drillholes are well spaced across the Kamoa 3 mining area.
- Some bias is expected on the number of joint sets that will be intersected by the drillholes. Normally not all the vertical joint sets are intersected by vertical drillholes.
- The stereographic analysis revealed two distinct joint sets in each rock type.
- Based on experience from Kamoa 1 and Kansoko, Kamoa 3 is expected to have at least three (3) Joint Sets in Diamictite and Siltstone and four (4) Joint Sets in the Sandstone.

It is recommended to continue structural logging as the underground mine is being developed to confirm ground support requirements.

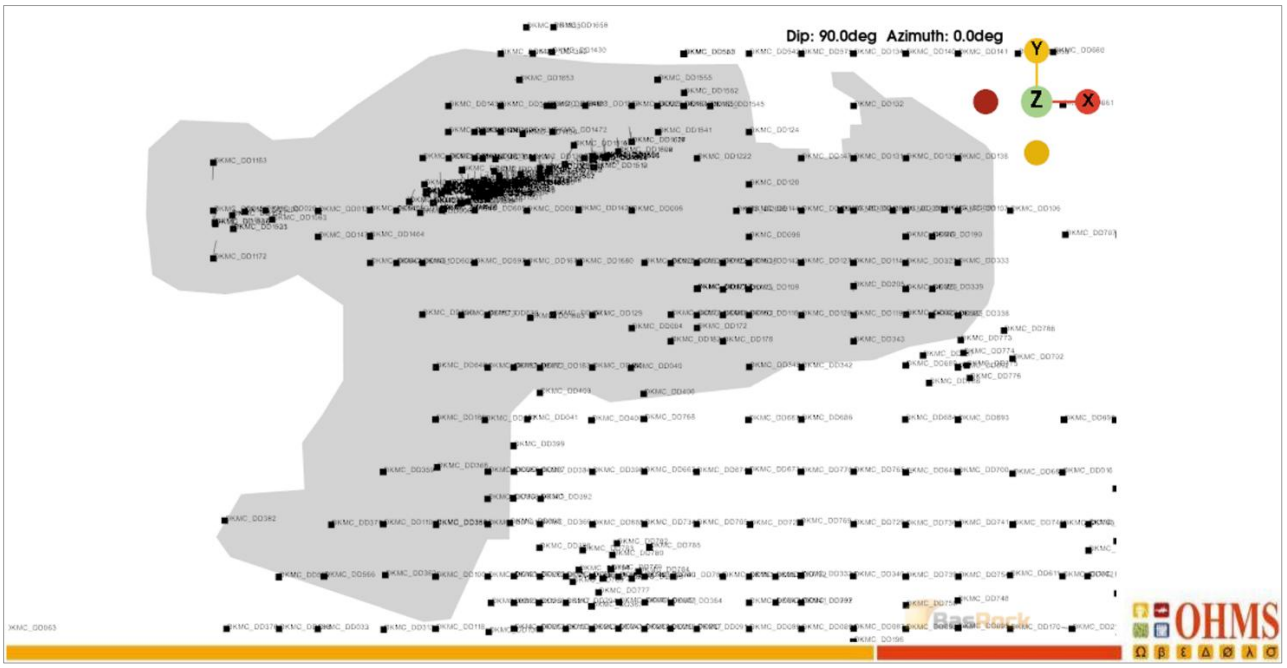
Table 16.14 Kamoa 3 structure sets

Lithology	Parameter	Joint Set 1	Joint Set 2	Joint Set 3	Joint Set 4	Joint Set 5	Joint Set 6
Diamictite (SDT)	Dip / Dip direction	17°/152°	88°/284°	-	-	-	-
	Number of joints	501	168	-	-	-	-
Siltstone (SSL)	Dip / Dip direction	16°/115°	89°/279°	-	-	-	-
	Number of joints	261	85	-	-	-	-
Sandstone (SST)	Dip	12°/121°	-	52°/011°	-	-	-
	Number of joints	76	-	17	-	-	-

16.2.5.7 Kamoa 4 geotechnical data summary

The geotechnical gap analysis (OHMS, 2025) of the Kamoa area indicates that the density of the geotechnical drillholes are wide spaced with fairly even coverage as shown in Figure 16.34.

Figure 16.34 Kamoa 4 geotechnical drillhole plan



Source: OHMS, 2025

Based on the RMR obtained from geotechnical logging, the rock mass conditions for the three main rock types range as follows:

- Diamictite ‘very poor’ to ‘good’ rock mass conditions (SDT)
- Siltstone ‘very poor’ to ‘good’ rock mass conditions (SSL)
- Sandstone ‘very poor’ to ‘good’ rock mass conditions (SST)

Rock mass characterization data for Kamoa 4 is summarized in Table 16.15.

Table 16.15 Kamoa 4 rock mass characterization summary

Material property	Statistical Distribution	RQD	RMR	GSI
SDT	25th percentile	66	44	39
	50th percentile	83	52	47
	75th percentile	95	62	57
	Average	**77**	**51**	**45**
SSL	25th percentile	37	38	33
	50th percentile	64	49	44
	75th percentile	85	57	52
	Average	**59**	**49**	**41**
SST	25th percentile	45	43	38
	50th percentile	77	53	48
	75th percentile	90	59	54
	Average	**67**	**51**	**46**

Note: adapted OHMS, 2025

Kamoa 4 laboratory testing material properties (Table 16.16) from the three main rock types are based on the following laboratory testing:

- Density (kg/m^3)
- Uniaxial Compressive Strength (MPa)
- Young's Modulus (GPa)
- Poisson's Ratio

Laboratory test samples are based on four drillholes with limited spread across the mine footprint:

- Only UCM and UTB testing were conducted on the selected samples inside the Kamoa 4 mining area.
- Based on the TLDC assessment more testing is needed.
- It is also recommended to perform TCS tests to gain insight into the effects of confinement on the rock's strength.

Table 16.16 Kamoa 4 laboratory testing summary

Material property	Rock unit	SDT	SSL	SST
Density (kg/m^3)	Number of tests	3	2	2
	Minimum	2670	2250	2040
	Mean	**2720**	**2455**	**2255**
	Maximum	2771	2660	2470
	Standard deviation	50	290	304
UCS (MPa)	Number of tests	3	2	2
	Minimum	19	8	8
	Mean	**73**	**15**	**22**
	Maximum	158	21	35
	Standard deviation	75	9	19
Young's modulus (GPa)	Number of tests	3	2	2
	Minimum	9.628	4.153	4.152
	Mean	**31.787**	**7.293**	**10.767**
	Maximum	65.100	10.432	17.382
	Standard deviation	29.370	4.440	9.355
Poisson's ratio	Number of tests	3	2	2
	Minimum	0.17	0.23	0.10
	Mean	**0.19**	**0.24**	**0.17**
	Maximum	0.25	0.25	0.23
	Standard deviation	0.05	0.02	0.09

Additional laboratory tests recommended by OHMS, 2025 are listed in the Table 16.17.

Table 16.17 Kamoa 4 laboratory testing recommendations for further work

Rock types	Test type	Number of current tests	Number of additional tests
SDT	TCS	0	24
	UCM	3	0
	UTB	2	14
SSL	TCS	0	24
	UCM	2	0
	UTB	2	14
SST	TCS	0	24
	UCM	2	0
	UTB	2	14

Kamoa 4 structural logging is listed in Table 16.18 and summarized below:

- The density of the structural drillholes are not well spaced across the Kamoa 4 mining area.
- Some bias is expected on the number of joint sets that will be intersected by the drillholes. Normally not all the vertical joint sets are intersected by vertical drillholes.
- The stereographic analysis revealed two distinct joint sets in each rock type with more or less the same orientations. One (1) flat dipping set and one (1) sub horizontal set.
- Structural logging is recommended as the underground mine is being developed.
- Ongoing structural logging will be required as the underground mine is being developed to confirm ground support requirements.

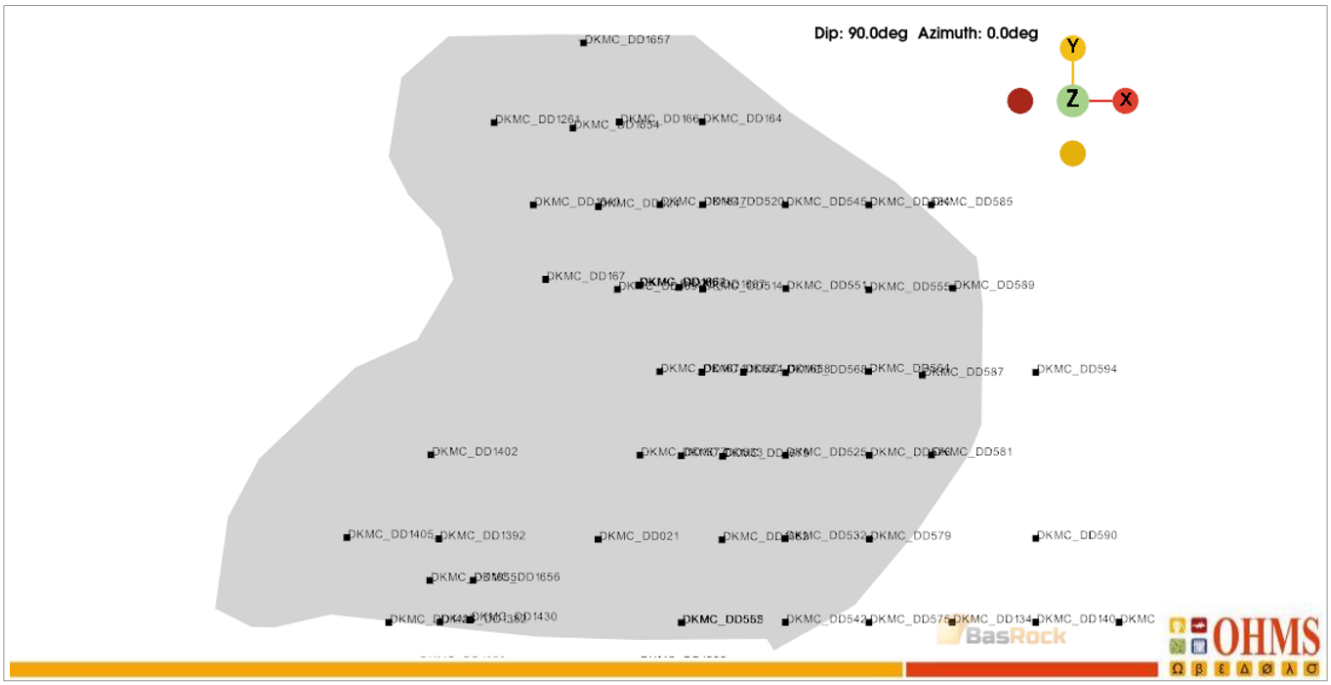
Table 16.18 Kamoa 4 structure sets

Lithology	Parameter	Joint set 1	Joint set 2	Joint set 3	Joint set 4	Joint set 5	Joint set 6
Diamictite (SDT)	Dip / Dip direction	64°/175°	6°/290°	-	-	-	-
	Number of joints	972	209	-	-	-	-
Siltstone (SSL)	Dip / Dip direction	55°/175°	3°/264°	-	-	-	-
	Number of joints	900	641	-	-	-	-
Sandstone (SST)	Dip	59°/174°	4°/220°	-	-	-	-
	Number of joints	170	161	-	-	-	-

16.2.5.8 Kamoa 5 geotechnical data summary

The geotechnical gap analysis (OHMS, 2025) of the Kamoa 5 area indicates that the density of the geotechnical drillholes are wide spaced with fairly even coverage as shown in Figure 16.35.

Figure 16.35 Kamoa 5 geotechnical drillhole plan



Source: OHMS, 2025

Based on the RMR obtained from geotechnical logging, the rock mass conditions for the three main rock types range as follows:

- Diamictite ‘poor’ to ‘good’ rock mass conditions (SDT)
- Siltstone ‘poor’ to ‘good’ rock mass conditions (SSL)
- Sandstone ‘poor’ to ‘good’ rock mass conditions (SST)

Rock mass characterization data for Kamoa 5 is summarized in Table 16.19.

Table 16.19 Kamoa 5 rock mass characterization summary (adapted OHMS, 2025)

Material property	Statistical distribution	RQD	RMR	GSI
SDT	25th percentile	60	44	39
	50th percentile	85	52	47
	75th percentile	97	59	54
	Average	**74**	**51**	**46**
SSL	25th percentile	48	42	37
	50th percentile	68	49	44
	75th percentile	92	57	52
	Average	**66**	**49**	**44**
SST	25th percentile	52	45	40
	50th percentile	71	51	46
	75th percentile	93	57	52
	Average	**68**	**51**	**46**

No laboratory testing for material properties on the three main rock types at Kamoa 5 has been conducted to date.

Laboratory tests recommended by OHMS, 2025 are listed in the Table 16.20.

Table 16.20 Kamoa 5 laboratory testing recommendations for further work

Rock types	Test type	Number of current tests	Number of additional tests
SDT	TCS	-	24
	UCM	-	8
	UTB	-	16
SSL	TCS	-	24
	UCM	-	8
	UTB	-	16
SST	TCS	-	24
	UCM	-	8
	UTB	-	16

Kamoa 5 structural logging is listed in Table 16.21 and summarized below:

- Only two (2) drillholes were structurally logged inside the Kamoa 5 mining area.
- Some bias is expected on the number of joint sets that will be intersected by the drillholes. Normally not all the vertical joint sets are intersected by vertical drillholes.
- The stereographic analysis revealed two (2) distinct joint sets in the Diamictite and Siltstone. Only one (1) joint set was recorded in the Sandstone.
- It is recommended to continue structural logging as the underground mine is being developed.

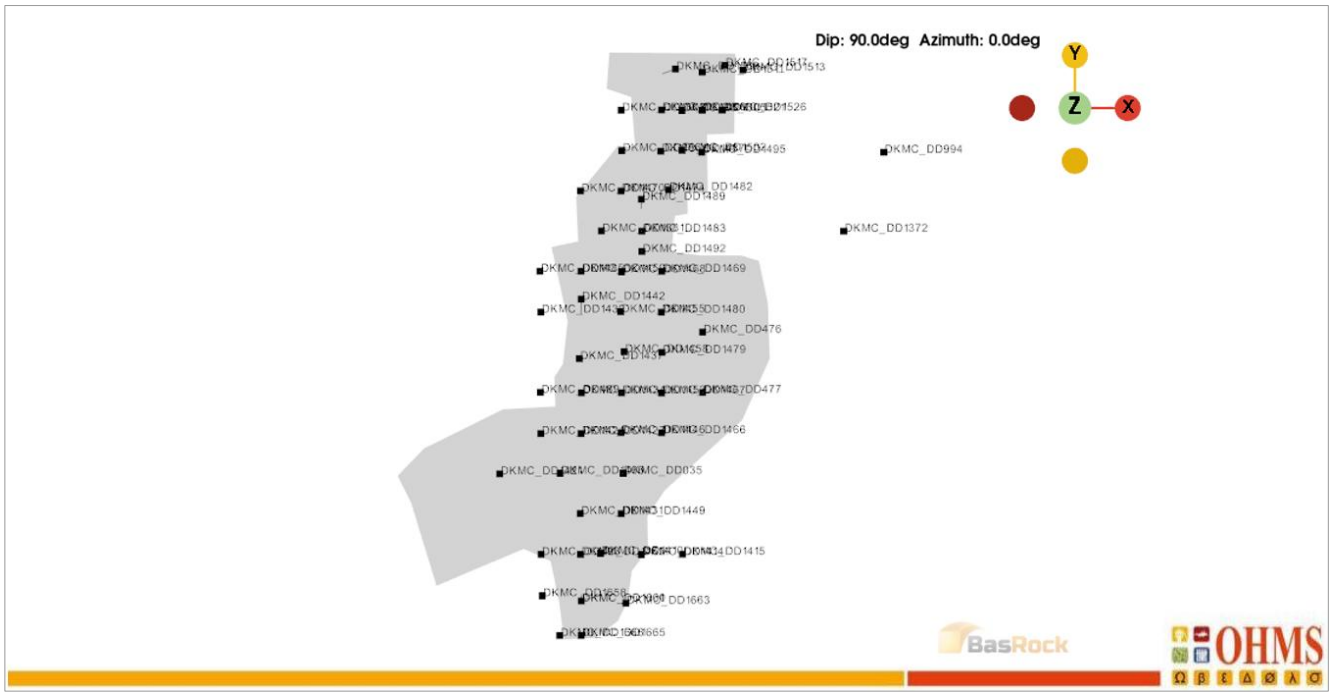
Table 16.21 Kamoa 5 structure sets

Lithology	Parameter	Joint set 1	Joint set 2	Joint set 3	Joint set 4	Joint set 5	Joint set 6
Diamictite (SDT)	Dip / Dip direction	4°/203°	88°/112°	-	-	-	-
	Number of joints	27	6	-	-	-	-
Siltstone (SSL)	Dip / Dip direction	5°/110°	-	50°/264°	-	-	-
	Number of joints	99	-	10	-	-	-
Sandstone (SST)	Dip	4°/117°	-	-	-	-	-
	Number of joints	8	-	-	-	-	-

16.2.5.9 Kamoa 6 geotechnical data summary

The geotechnical gap analysis (OHMS, 2025) of the Kamoa 6 area indicates that the density of the geotechnical drillholes are wide spaced with even coverage as shown in Figure 16.36.

Figure 16.36 Kamoa 6 geotechnical drillhole plan



Source: OHMS, 2025

Based on the RMR obtained from geotechnical logging, the rock mass conditions for the three main rock types range as follows:

- Diamictite ‘very poor’ to ‘good’ rock mass conditions (SDT)
- Siltstone ‘very poor’ to ‘good’ rock mass conditions (SSL)
- Sandstone ‘poor’ to ‘fair’ rock mass conditions (SST)

Rock mass characterization data for Kamoa 6 is summarized in Table 16.22.

Table 16.22 Kamoa 6 rock mass characterization summary (adapted OHMS, 2025)

Material property	Statistical distribution	RQD	RMR	GSI
SDT	25th percentile	58	34	29
	50th percentile	88	53	48
	75th percentile	100	67	62
	Average	**74**	**49**	**44**
SSL	25th percentile	18	33	28
	50th percentile	50	45	40
	75th percentile	79	54	49
	Average	**48**	**42**	**37**
SST	25th percentile	48	45	40
	50th percentile	59	49	44
	75th percentile	69	49	44
	Average	**55**	**47**	**42**

No laboratory testing for material properties on the three main rock types at Kamoa 6 has been conducted to date.

Laboratory tests recommended by OHMS, 2025 are listed in the Table 16.23.

Table 16.23 Kamoa 6 laboratory testing recommendations for further work

Rock types	Test type	Number of current tests	Number of additional tests
SDT	TCS	-	24
	UCM	-	8
	UTB	-	16
SSL	TCS	-	24
	UCM	-	08
	UTB	-	146
SST	TCS	-	24
	UCM	-	8
	UTB	-	16

Kamoa 6 structural logging is listed in Table 16.24 and summarized below:

- Only five (5) drillholes were structurally logged inside the Kamoa 6 mining area. Three (3) in the North and two (2) in the south. No drilling in the central part of Kamoa 6.
- Some bias is expected on the number of joint sets that will be intersected by the drillholes. Normally not all the vertical joint sets are intersected by vertical drillholes.
- The stereographic analysis revealed one (1) distinct joint set in each rock type with more or less the same orientation.
- It is recommended to continue structural logging as the underground mine is being developed.

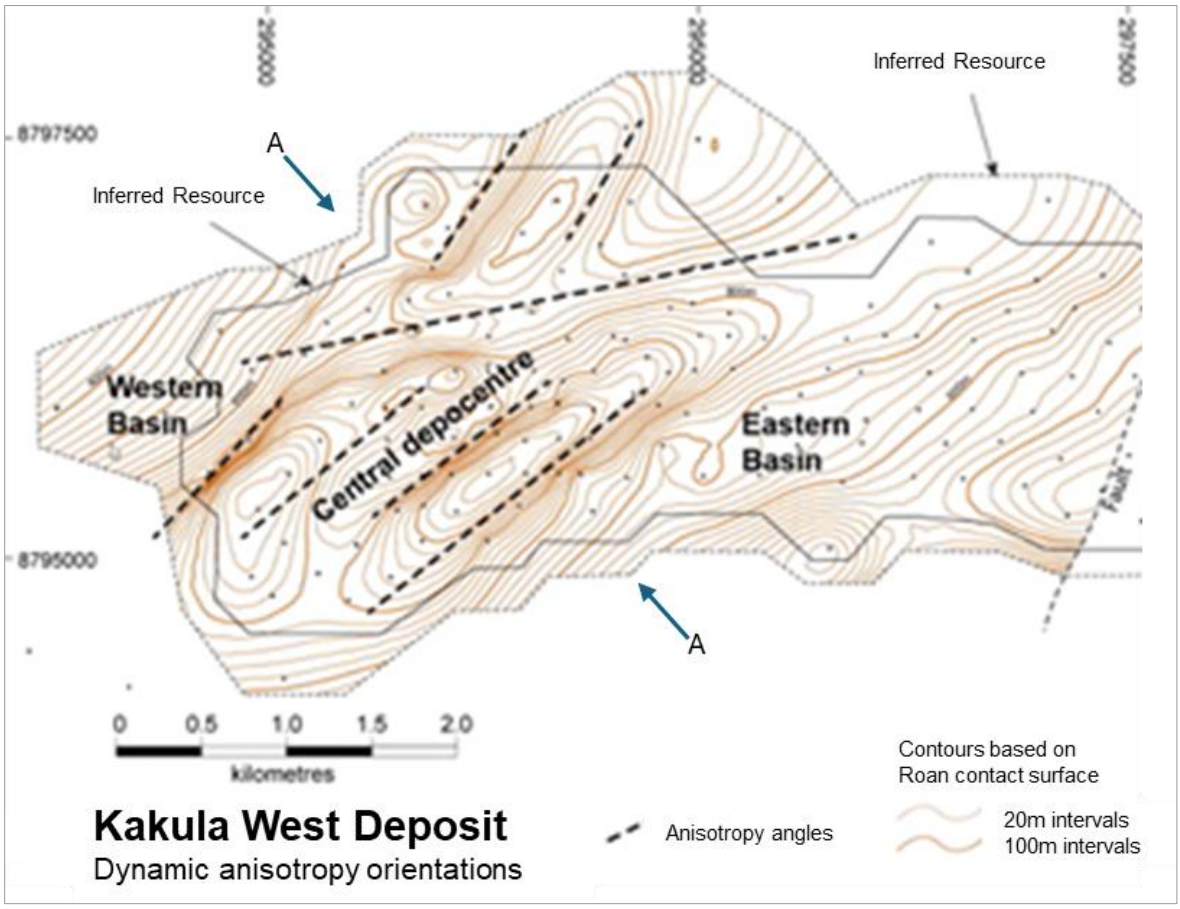
Table 16.24 Kamoa 6 structure sets

Lithology	Parameter	Joint set 1	Joint set 2	Joint set 3	Joint set 4	Joint set 5	Joint set 6
Diamictite (SDT)	Dip / Dip direction	9°/209°	-	-	-	-	-
	Number of joints	35	-	-	-	-	-
Siltstone (SSL)	Dip / Dip direction	3°/302°	-	-	-	-	-
	Number of joints	72	-	-	-	-	-
Sandstone (SST)	Dip	6°/102°	-	-	-	-	-
	Number of joints	22	-	-	-	-	-

16.2.5.10 Kakula West geotechnical data summary

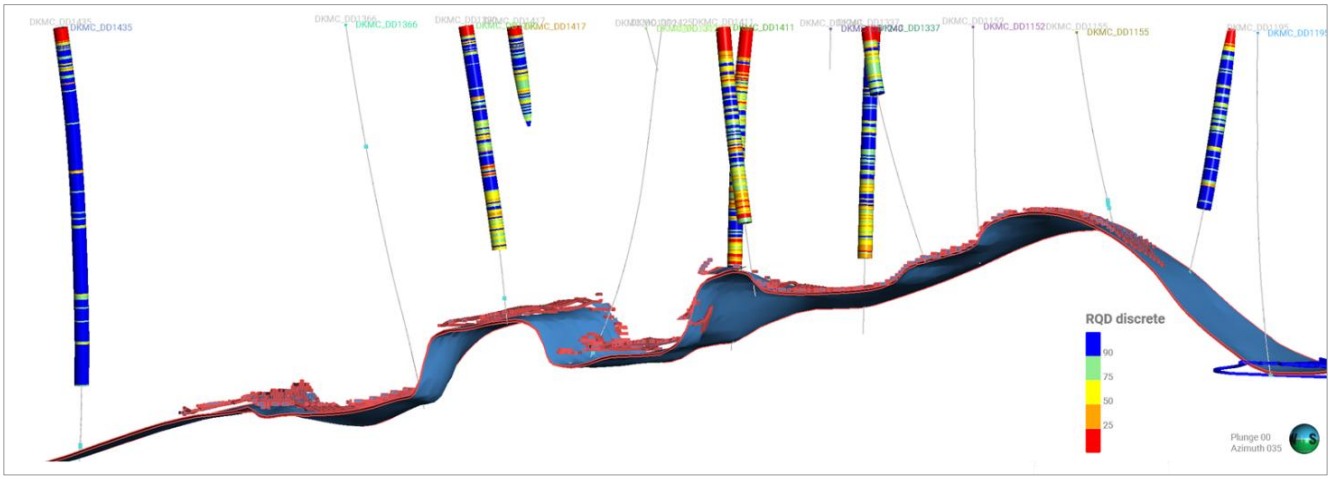
It is evident from the Kakula West basin formations in Figure 16.37, that there are larger areas of poor, and very-poor, ground conditions when compared to Kakula, and it is structurally more complex (Figure 16.38).

Figure 16.37 Kakula West deposit



Source: Ivanhoe, 2025

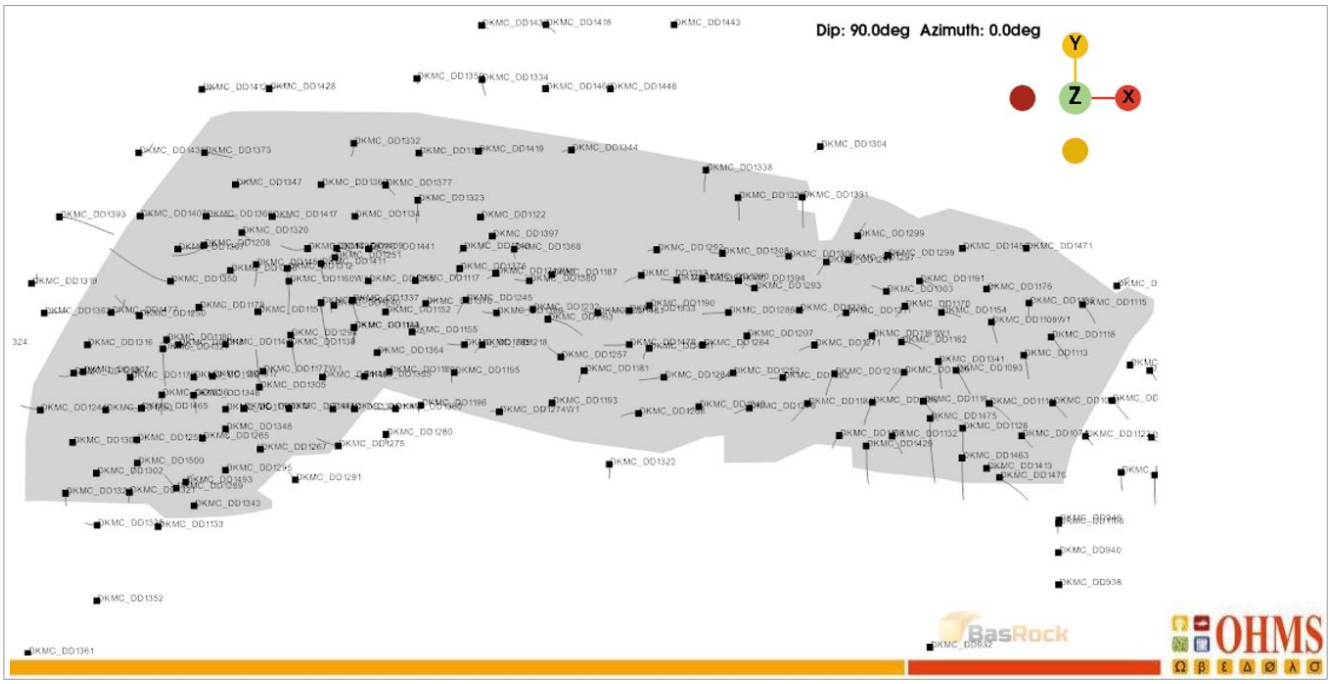
Figure 16.38 Kakula West deposit looking northeast showing RQD



Source: Ivanhoe 2025

The geotechnical gap analysis (OHMS, 2025) indicates that the density of the geotechnical drillholes are well spaced across the Kakula West mining area shown in Figure 16.39. All future data collection should ensure that geotechnical data is collected into the footwall sandstone.

Figure 16.39 Kakula West geotechnical drillhole plan



Source: OHMS, 2025

Based on the RMR values obtained from geotechnical logging, the rock mass conditions for the three main rock types range as follows:

- Diamictite ‘poor’ to ‘good’ rock mass conditions (SDT)
- Siltstone ‘poor’ to ‘good’ rock mass conditions (SSL)
- Sandstone ‘poor’ to ‘good’ rock mass conditions (SST)

Rock mass characterization data for Kakula West is summarized in Table 16.25.

Table 16.25 Kakula West rock mass characterization summary (adapted OHMS, 2025)

Material property	Statistical distribution	RQD	RMR	GSI
SDT	25th percentile	85	56	51
	50th percentile	97	61	56
	75th percentile	100	84	79
	Average	**88**	**65**	**60**
SSL	25th percentile	55	43	38
	50th percentile	78	61	56
	75th percentile	99	67	62
	Average	**69**	**59**	**54**
SST	25th percentile	80	55	50
	50th percentile	90	61	56
	75th percentile	100	85	80
	Average	**89**	**68**	**63**

Kakula West laboratory testing material properties (Table 16.26) from the three main rock types are based on the following laboratory testing:

- Density (kg/m^3)
- Uniaxial Compressive Strength (MPa)
- Young's Modulus (GPa)
- Poisson's Ratio

Laboratory test samples are based on twelve drillholes with generally well-spaced spread in across the mine footprint:

- Only UCM and UTB testing were conducted on the selected samples inside the Kakula West mining area.
- Based on the TLDC assessment more testing is needed, particularly out towards the extent of the mine footprint.
- It is also recommended to perform TCS tests to gain insight into the effects of confinement on the rock's strength.
- Samples should also be taken in the Kamoa Pyritic Siltstone (KPS) and Breccia (BRE) rock types to ensure data is available for development in these rock types.

Table 16.26 Kakula West laboratory testing summary

Material property	Rock unit	SDT	SSL	SST	SDT/BRE	SDT/SSL	SST/SSL
Density (kg/m^3)	Number of tests	19	21	-	3	6	3
	Minimum	1960	2330	-	2650	2040	2580
	Mean	**2418**	**2541**	**-**	**2727**	**2400**	**2597**
	Maximum	2700	2660	-	2780	2760	2620
	Standard deviation	282	76	-	68	363	21
UCS (MPa)	Number of tests	8	4	1	1	1	1
	Minimum	42	34	54	68	52	25
	Mean	**68**	**54**	**54**	**68**	**52**	**25**
	Maximum	118	75	54	68	52	25
	Standard deviation	23	17	-	-	-	-
Young's modulus (GPa)	Number of tests	8	4	1	1	1	1
	Minimum	17.746	10.081	55.261	40.653	68.137	19.787
	Mean	**57.833**	**34.128**	**55.261**	**40.653**	**68.137**	**19.787**
	Maximum	114.483	64.327	55.261	40.653	68.137	19.787
	Standard deviation	32.839	24.510	-	-	-	-
Poisson's ratio	Number of tests	8	4	1	1	1	1
	Minimum	0.14	0.15	0.35	0.24	0.23	0.23
	Mean	**0.21**	**0.18**	**0.35**	**0.24**	**0.23**	**0.23**
	Maximum	0.26	0.22	0.35	0.24	0.23	0.23
	Standard deviation	0.04	0.03	-	-	-	-

Additional laboratory tests recommended by OHMS, 2025 are listed in the Table 16.27.

Table 16.27 Kakula West laboratory testing recommendations for further work

Rock types	Test type	Number of current tests	Number of additional tests
SDT	TCS	0	24
	UCM	28	0
	UTB	20	16
SSL	TCS	0	24
	UCM	7	1
	UTB	0	16
SST	TCS	0	24
	UCM	8	0
	UTB	0	16

Kakula West structural logging is listed in Table 16.28 and summarized below:

- The density of the structural drillholes is well spaced across the Kakula West mining area.
- Some bias is expected on the number of joint sets that will be intersected by the drillholes. Normally not all the vertical joint sets are intersected by vertical drillholes.
- Stereographic analyses indicate four distinct joint sets in each both the Diamictite and Siltstone and two in the Sandstone.
- Based on experience from Kakula, it is expected that Kakula West will have at least three (3) Joint Sets in Diamictite and Siltstone and four (4) Joint Sets in the Sandstone.
- Structural logging should be ongoing as the underground mine is being developed.

Table 16.28 Kakula West structure sets

Lithology	Parameter	Joint set 1	Joint set 2	Joint set 3	Joint set 4	Joint set 5	Joint set 6
Diamictite (SDT)	Dip / Dip direction	63°/342°	54°/089°	5°/082°	63°/138°	-	-
	Number of joints	622	272	172	98	-	-
Siltstone (SSL)	Dip / Dip direction	57°/336°	57°/076°	5°/120°	59°/121°	-	-
	Number of joints	55	81	240	-96	-	-
Sandstone (SST)	Dip	51°/331°	-	21°/331°	55°/121°	-	-
	Number of joints	21	-	128	152	-	-

16.2.6 Support requirements

Ground support deficiencies and operational discipline, along with other factors, were a contributing factor to the consequences of the failure incident at Kakula Copper Mine in late May 2025.

Key factors to ensure appropriate ground support regimes in the future include:

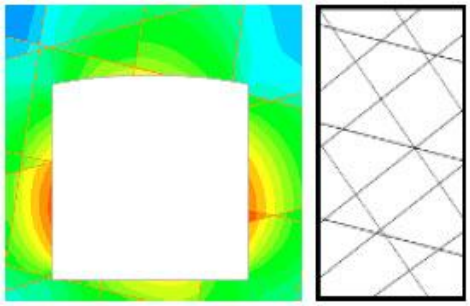
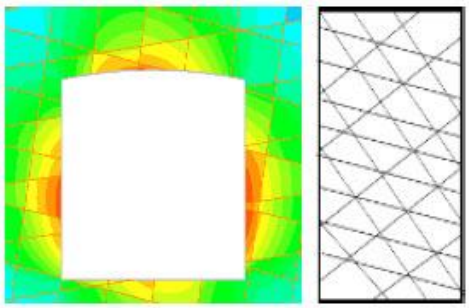
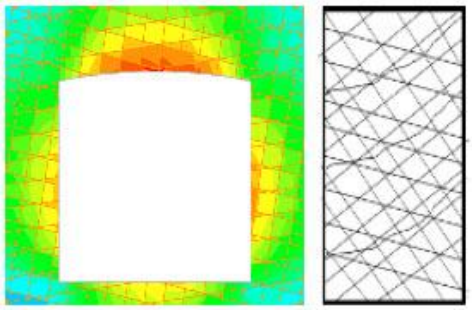
- Oriented infill drillholes across each mine.
- Updated geotechnical data management systems for rock mass characterization of lithological units and to address inconsistencies and gaps.
- Geotechnical mapping on development to collect data on structure persistence (trace length) of the geological discontinuities, as they control the size of the unstable geometries that can form at the exposed spans of the excavations.

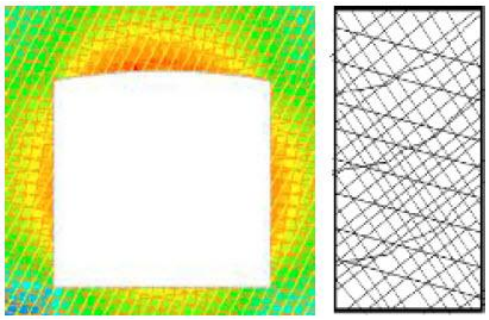
- Geotechnical planning, risk management and design input that align with appropriate mining methods and production priorities.
- Updated modelling as new data becomes available for calibration against observed conditions (rockfalls, pillar damage, or subsidence) and dynamic loading scenarios for interaction between multiple stoping blocks, particularly extraction ratios greater than 60%.

16.2.6.1 Support design considerations

Primary support must cater for the local rock mass conditions. The geotechnical data is evaluated according to the Tunnel Quality Index 'Q' (Barton, 1974) rock mass classification system. Five rock mass quality classes (Q) are used to guide ground support recommendations as shown in Table 16.29.

Table 16.29 Q Rating Ground support type

GCD	Ground conditions		Q Rating	Ground support type
1		Massive or slightly fractured rock (Three Joints Set Number with low Joint Frequency-FF/m=>5 m). Reef dip 0° to 10°. End mined in Diamictite. Q = Good. Normal blast round length.	> 10-40 GOOD	Type 1
2		Massive or slightly fractured rock (Three Joints Set Number with low Joint Frequency-FF/m=>5 m). Reef dip 0° to 10°. End mined in Diamictite. Q = Fair. Normal blast round length.	≥ 7 and ≤9.9 FAIR	Type 2
3		Moderately fractured rock. (Three Joints Set Number with Poor Joint Frequency-FF/m=2.5-1 m). Reef dip from 10° to 35°. Hanging wall in Diamictite and Siltstone in sidewalls. Q = Poor.	≥ 4 and ≤6.9 POOR	Type 2

GCD	Ground conditions		Q Rating	Ground support type
4		Crushed or brecciated rock (four or more joint sets, very heavily jointed with very high Joint Frequency FF/m=<0.5 m). Reef dip from 10° to 35°. Q=Very Poor Reduced blast round length and reduced heading size, trim blasting. Highly stresses areas.	≥1 and ≤3.9 VERY POOR	Type 3
5	All long-term excavations (Life span of 5 years and longer), include access drives, declines and workshops.			Type 4

16.2.7 Ground support recommendations

The current ground support requirements have been determined based on consideration of the empirical assessment presented above and are provided in the 'All Kamoa Copper Support Standards Book' which are summarized in Table 16.30.

Table 16.30 Summary of All Kamoa Copper Support Standards Book, 2025-02-18

Support type	Profile size	Rock mass rating (Q)		Support standard*	
1A	Stoping Standard 6 m to 7.5 m wide 7.5 m (max) high	>10-40	Good Conditions	Primary	- 2.4 m S420 Mild Steel Cambolt Split Sets, with dual purpose washer spaced on a 1.5 m x 1.3 m (ring spacing) in the hangingwall. - 2.4 m S420 Mild Steel Cambolt Split Sets, with W-plate combi washers spaced on a 1.5 m x 1.3 m (ring spacing) in sidewalls.
	Developing Standard 6 m to 7.5 m wide 7.5 m (max) high	≥ 7 and ≤ 9.9	Fair Conditions		- 2.4 m 3CR12 Cambolt Split Sets, with dual purpose washer spaced on a 1.5 m x 1.3 m (ring spacing) in the hangingwall. - 2.4 m 3CR12 Cambolt Split Sets, with W-plate combi washers spaced on a 1.5 m x 1.3 m (ring spacing) in sidewalls.
1B	Stoping Standard 9 m wide 10 m (max) high	>10-40	Good Conditions	Primary	- 3.0 m 3CR12 Cambolt Split Sets, with dual purpose washer spaced on a 1.5 m x 1.3 m (ring spacing) in the hangingwall. - 2.4 m S420 Mild Steel Cambolt Split Sets, with W-plate combi washers spaced on a 1.5 m x 1.3 m (ring spacing) in sidewalls.
	Developing Standard 9 m wide 10 m (max) high	≥ 7 and ≤ 9.9	Fair Conditions		- 3.0 m 3CR12 Cambolt Split Sets, with dual purpose washer spaced on a 1.5 m x 1.3 m (ring spacing) in the hangingwall. - 2.4 m S420 Mild Steel Cambolt Split Sets, with W-plate combi washers spaced on a 1.5 m x 1.3 m (ring spacing) in sidewalls.
2A	Stoping Standard 6 m to 7.5 m wide 7.5 m (max) high	≥ 4 and ≤ 6.9	Poor Conditions	Primary	- 2.4 m S420 Mild Steel Cambolt Split Sets, with dual purpose washer spaced on a 1.5 m x 1.0 m (ring spacing) in the hangingwall. - 2.4 m S420 Mild Steel Cambolt Split Sets, with W-plate combi washers spaced on a 1.5 m x 1.0 m (ring spacing) in sidewalls.
	Developing Standard 6 m to 7.5 m wide 7.5 m (max) high				- 2.4 m 3CR12 Cambolt Split Sets, with dual purpose washer spaced on a 1.5 m x 1.0 m (ring spacing) in the hangingwall. - 2.4 m 3CR12 Cambolt Split Sets, with W-plate combi washers spaced on a 1.5 m x 1.0 m (ring spacing) in sidewalls.

Support type	Profile size	Rock mass rating (Q)		Support standard*	
2B	Stoping Standard 9 m wide 10 m (max) high	≥ 4 and ≤ 6.9	Poor Conditions	Primary	- 3.0 m 3CR12 Cambolt Split Sets, with dual purpose washer spaced on a 1.5 m x 1.0 m (ring spacing) in the hangingwall. - 2.4 m S420 Mild Steel Cambolt Split Sets, with W-plate combi washers spaced on a 1.5 m x 1.0 m (ring spacing) in sidewalls.
	Developing Standard 9 m wide 10 m (max) high				- 3.0 m 3CR12 Cambolt Split Sets, with dual purpose washer spaced on a 1.5 m x 1.0 m (ring spacing) in the hangingwall. - 2.4 m 3CR12 Cambolt Split Sets, with W-plate combi washers spaced on a 1.5 m x 1.0 m (ring spacing) in sidewalls.
3A	Stoping Standard 6 m wide 6 m (max) high	≥ 1 and ≤ 3.9	Very Poor Conditions	Primary	- 2.4 m S420 Mild Steel Cambolt Split Sets, with dual purpose washer spaced on a 1.0 m x 1.0 m (ring spacing) in the hangingwall. - 2.4 m 3 S420 Mild Steel Cambolt Split Sets, with W-plate combi washers spaced on a 1.0 m x 1.0 m (ring spacing) in sidewalls.
	Developing Standard 6 m wide 6 m (max) high				- 2.4 m 3CR12 Cambolt Split Sets, with dual purpose washer spaced on a 1.0 m x 1.0 m (ring spacing) in the hangingwall. - 2.4 m 3CR12 Cambolt Split Sets, with W-plate combi washers spaced on a 1.0 m x 1.0 m (ring spacing) in sidewalls. High-tensile galvanized welded mesh (5.6 mm, 1.0 m x 1.0 m) or W-plates (0.5 m x 0.5 m x 5mm) with reduced support spacing (0.8 m x 0.8 m). Alternatively, V-Seal or shotcrete applied to hangingwall and sidewall.
4A	Single and Double Breakaway – Curved Corner	All Intersections		Primary	- 3.0 m 3CR12 Cambolt with W-plate combi washers on a 1.5 m x 1.5 m spacing until 3 m pas the bull nose position in all directions. This is in addition to where 2.4 m Cambolts were installed. - 16 mm V-Seal / Mesh on both side walls and hangingwall based on Rock Engineering recommendations. - Install 5.5 m x 20 t end anchored cable anchors spaced on a 2.0 m block pattern based on life span of excavation until 3 m past the bull nose position in all directions. - Unwedge analysis where cross span is more than 20 m for required support.

Notes: *

- No water, if water intersected, contact Rock Engineering immediately.
- Unwedge analysis must be conducted when mining depth below surface is less than 100 m for intersection support recommendations.
- No siltstone to be left in the hangingwall when the ground conditions are problematic.
- Ensure support is installed as close as possible to 90 to the dip of reef / major joint set.

Ground support recommendations should consider:

- Industry practice is to use split sets, mesh, cablebolts and fibrecrete (if required) as a ground support system to retain fractured rock and prevent unravelling and deterioration under induced stress, not split sets alone.
- AMC recommends that surface support (weld mesh or fibrecrete) should be installed in the backs and shoulders of all development as a minimum.

- AMC recommends fully encapsulated bolts be considered for longer term development if wet conditions are encountered and for the ore zone. These bolts (with either cement grout or resin) provide protection against corrosion and have a higher capacity to support any brows that might be formed in the ore.
- V-seal is not an alternative to mesh. Mesh is considered appropriate for hangingwall and sidewalls in all ground conditions in line with mining industry standards.
- Structural analysis indicates the potential for gravity falling and wedge failure modes (at all depths). Systematic pattern cablebolting will be required in profiles greater than standard development width.
- Updated ground support standards will be necessary as the new mines are developed.
- Excavations should be scheduled to ensure the appropriate ground support is installed wherever possible, and appropriate for the expected life span of the opening.
- Systematic pattern cablebolting is required in wide spans, stopes and intersections. A review of geotechnical drilling and mapping will be required during short term planning to ensure that potential failure modes have been identified, and appropriate length ground support elements are designed and installed.
- Careful excavation of intersections is required to ensure spans are mined to design. Larger spans will require more support than indicated.
- Some zones of very poor ground are expected at Kakula West. Specialized development techniques such as ‘spiling’ (fore-poling), short rounds and the use of shotcrete arches are likely to be required to manage some of the poorer ground conditions and achieve adequate control of development stability.

16.2.8 Overall mine stability

Numerical modelling to confirm global stability of the mine designs and schedules is ongoing by Beck Engineering. Modelling of the Kakula mine failure, the causes, learnings, outcomes and design recommendations for overall mine stability are being implemented for each of the mines to address geotechnical risk related to global stability, including:

- **Mine design** - mining method, designs and schedules, processes for management of change.
- **Available data** - technical gaps in geotechnical data, data validation and oversimplification of parameters used in numerical models.
- **Ground support** - deficiencies, oversize spans, large open spans for long time periods leading to dynamic loading and progressive failure, implementation controls.
- **Monitoring** - seismic monitoring limitations for early detection of global stability, big picture, cumulative effect understanding, fall of ground and damage assessment.

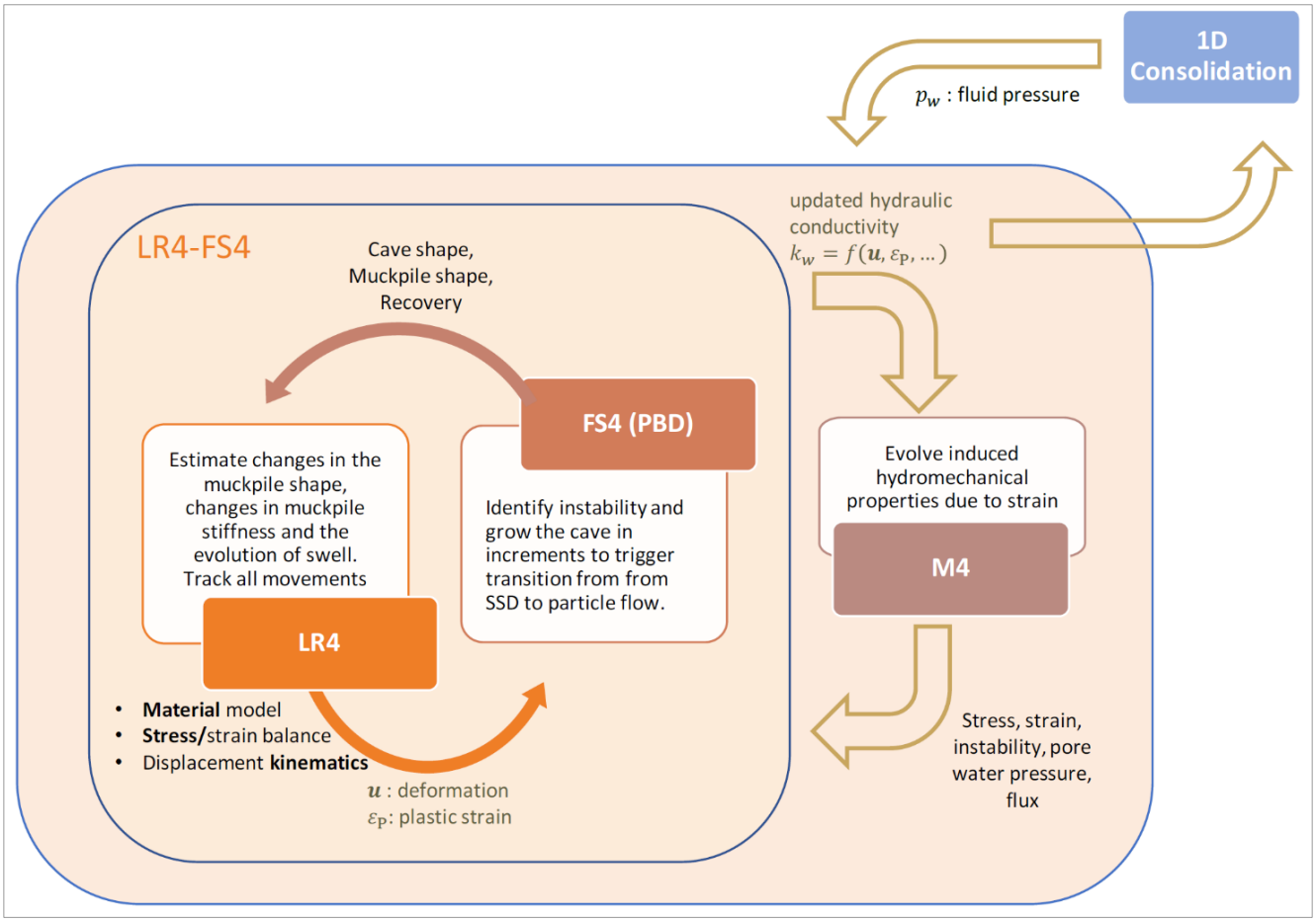
Beck (2025) used LR4-FS4 numerical modelling technique for this project and outlines that it comprises coupling of FE and DE solvers to simulate dilatant, strain softening, disassembly and flow of failed rock masses, back fill and transient fluid flow (Figure 16.40). In this case, the DE objects were not required and only the FE (LR4) parts were used.

Beck describes the selection of the FE framework because it is large strain and incorporates a 3D softening and dilation potential, which is necessary to simulate the connectedness in a system of pillars, including the post-failure response. The FE framework is also optimized for very large models. In this case, similitude requires many hundreds of development, stoping and filling steps and representation of the complete

geometry using higher order elements, leading to more than 100 million Degrees of Freedom before HM coupling.

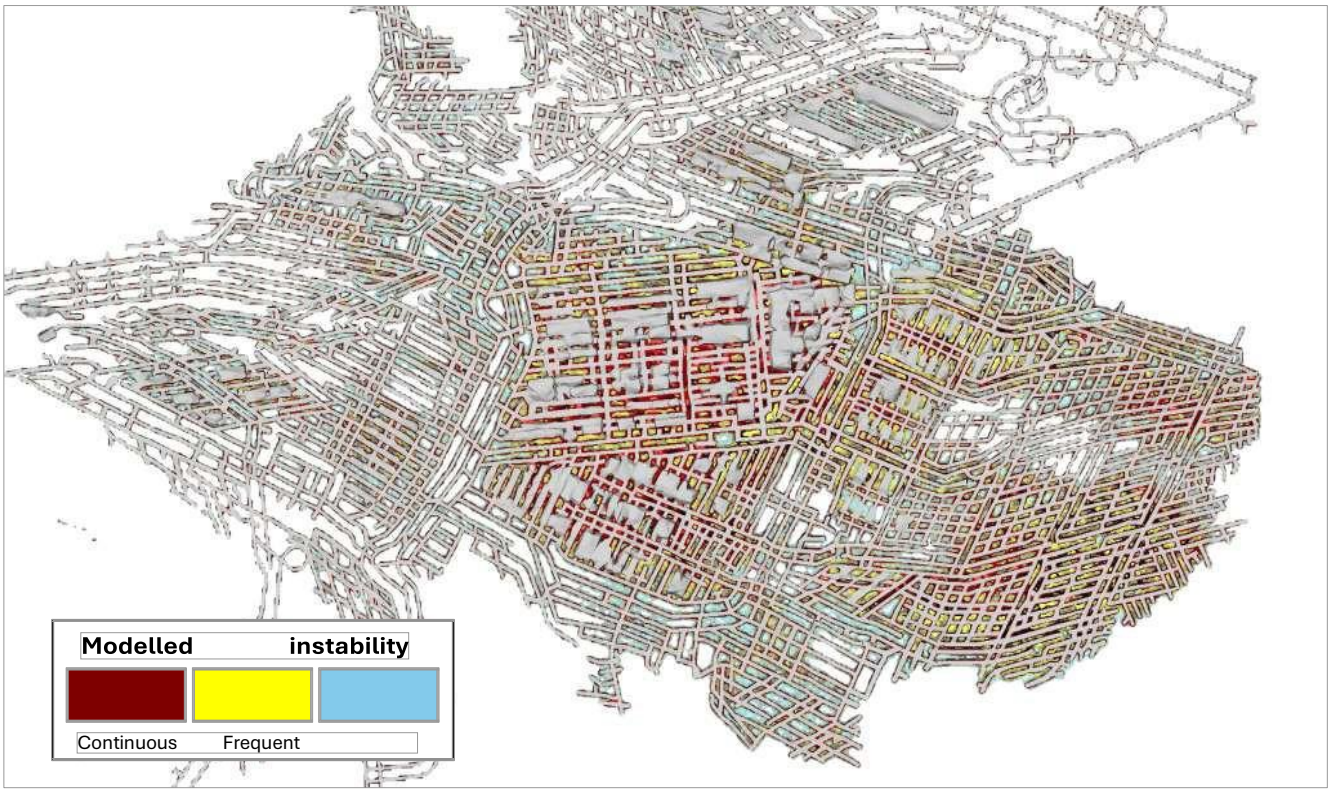
In summary, numerical modelling allowed a direct comparison between modelled and measurements surface movements as shown in Figure 16.41.

Figure 16.40 Overview of FE-FS4 cave coupling



Source: Beck, 2025
Note: The hydromechnical coupling and consolidation were not implemented in this project, but the forward analysis of the proposed mine plan will be hydromechnically coupled.

Figure 16.41 Modelled comparison at the Kaula mine



Source: Beck 2025
Note: A modelled comparison of simulated damage and the observed pattern of emerging, continuous and frequent damage available at this time at Kaula mine.

Based on the numerical modelling outcomes of the Kakula mine failure, the following mine design recommendations were provided for overall mine stability, layouts and geometry, including:

- Mining blocks (spans) will be reduced in size compared to previous designs.
- Major access pillars will be increased in dimension to enhance stability.
- Blocks will be subdivided into panels, with intervening pillars for separation.
- Ribs and chain pillars will be retained within panels to manage deformation.
- Block extraction will occur in staged phases, with progression contingent on the measured performance of each preceding stage; design modifications will be implemented as necessary based on observed outcomes.
- Development, stoping, and filling activities will advance concurrently under defined protocols, regulating the sequential progression of each cycle step to maintain operational balance; any slowdown in one area will correspondingly constrain the others to prevent systemic imbalances.
- Strict guidelines will govern the permissible extent of unfilled voids within each block, such that delays in filling operations will proportionally decelerate overall mining activities.
- Extraction sequences and schedules will be tightly controlled to ensure that premature over-extraction ahead of the main stoping front is prevented.
- Continuous monitoring of stope and tunnel performance, subsidence, deformation, and seismicity will verify that stability and deformation remain within expected parameters, facilitating responsive design adjustments based on monitoring data and lessons learned.

- Permanent accesses will be strictly dimensioned—no permanent access will involve floor stripping. Planning processes will be revised accordingly.
- The preliminary mine plan will emphasize stability, water management, and production targets, incorporating comprehensive exploration and monitoring measures.
- Global strategies and extraction sequences will be systematically evaluated and refined using calibrated global nonlinear models as part of long-term planning cycles.
- The draft mine plan will integrate realistic constraints and minimum requirements for extraction sequences, filling, ore and water handling, and exploration to mitigate operational imbalances.
- Lessons from prior events will be embedded into the plan via stringent controls as described above.
- During operations, global, block, and pillar scale performances and interactions will be analyzed against expected outcomes to proactively address emerging issues and capitalize on opportunities. Tactical adjustments to pillars and stopes will be employed to address adverse geological conditions or to manage excessive deformation.
- Plans, strategies, and expectations will undergo continuous review, informed by the latest geotechnical and geological data, and all available performance observations, allowing for adaptive modifications as required.
- Stability management will be supported by tools including:
  - The soon-to-be-installed seismic monitoring system.
  - Damage mapping and convergence scanning.
- These datasets will inform assessments of the stability and load distribution within the mine's pillar system. If stability is deemed insufficient upon review of measured data, stopes may be designated as tactical pillars and / or access / stabilization pillar sizes will be adjusted as necessary.

The extraction process will occur in stages:

- The first phase establishes key access to the initial block, services, and dewatering.
- The second phase involves possible in-block access corridors, which may be adjusted as new information emerges.
- Further development for stoping or main drifts will reflect updated knowledge and spacing.
- Secondary stoping or drifting will only proceed if conditions are suitable and performance meets expectations; sometimes these will not be feasible due to site conditions.
- In drift and fill (D&F) mining, abandoning some secondary or tertiary stopes is common.
- Design controls must align with the method statement and Ground Control plan and require regular audit for compliance.

A preliminary guide to inform scheduling and design is provided in Section 16.2.12.1. This guide is flexible and subject to further analysis based on orebody topology and lessons learned from past events.

16.2.9 Operational implementation

The design and schedules for each mine have been developed with lessons learnt from Kakula mine and the work undertaken is a guide, subject to ongoing data collection and analysis to ensure flexibility with the variability of the orebody and rock mass conditions.

For operational implementation, the design control measures must be integrated with the method statement, and the mines Ground Control management plan, and audited for efficacy and compliance on a regular basis to manage geotechnical risk.

The recommendations provided in this section refer to further work required for operational implementation.

16.2.9.1 Data collection

The available geotechnical drillholes with logged parameters suitable for empirical modelling were reviewed, gap analysis undertaken and recommendations for further requirements have been provided. When the targets for underground mining are better defined (from mine design studies), additional drillholes locations should be selected to improve the understanding of the ground conditions.

AMC recommends that additional geotechnical data be collected from any additional diamond drillholes that intersect the mine designs, such as infill drilling. The data collection programme should include:

- Geotechnical logging of additional drillholes through the proposed underground mining area. The mine design will need to ensure flexibility for change as required.
- Conduct rock laboratory tests to determine strength and elastic parameters for the underground, especially for the materials / lithologies where the current data is limited or absent
- Conduct a stress measurement testing for input into numerical modelling.
- Site specific investigations are recommended to investigate ground conditions at the proposed boxcut sites, and for any raise boreholes (e.g. ventilation shafts) to confirm locations. Near surface conditions may be affected by weathering and this is often associated with poorer ground conditions and stability issues.
- A geotechnical cover hole for decline advance is recommended to support planned increased development rates, with this task being decoupled from development activities.

AMC recommends that additional geotechnical data be collected from the current pit. The data collection programme should include:

- Geotechnical mapping of the various mining areas including structure orientation, characteristics, trace length and persistence, spacing.
- Mapping of large-scale structures.

16.2.9.2 Update assessments

The additional geotechnical data should be used to confirm the geotechnical domains and update or confirm the supporting information including:

- Update the rock mass classification, stope spans, and ground support assessments based on the new information.
- Assessment for in situ stresses.
- Assessment of minor structures, wedge analyses.
- Models of faults and large-scale structures which could impact development or stoping.
- Define the material properties and conduct numerical modelling to assess the stability of long-term infrastructure, pillars, drifts, drift and fill stoping sequence at both operational and global scales.

The assessments should be revised to a higher level of detail using the updated information. This would include:

- Stope stability and pillar assessments according to local ground conditions, including the potential impact of large-scale structures.
- Geotechnical assessment of the portal(s), once a location has been determined.
- Geotechnical assessment of any planned raise bore shafts.
- Consideration should be given in mine design to the location of poor ground associated with possible faults for the placement of infrastructure. Large spans should be avoided in these areas, and development should be oriented perpendicular to faults.
- Once available, conduct numerical modelling of the mine designs to determine the mining sequence, infrastructure stability, pillar and backfill stability.

16.2.9.3 Monitoring

Once mining commences, a geotechnical monitoring programme should be implemented and include:

- Routine visual monitoring is recommended for all operating drifts, stopes and development to monitor deterioration of the rock mass and ground support.
- Seismic monitoring system to assess global stability precursors.
- Some instrumentation may be required to optimize support systems, particularly stope reinforcement and for capital infrastructure.

16.2.9.4 Blasting

Smooth-wall blasting is recommended for all underground development. The zone of loosening caused by blasting is typically 0.3 m to 0.5 m into the rock with good blasting practices. This can increase up to 0.8 m with poor blasting practices which in turn increases the ground support requirements.

Routine down-hole surveys are recommended for stope drillholes as part of a quality control programme.

16.2.9.5 Scaling

Scaling is recommended in underground development as routine practice, particularly for the lower walls where no surface support is installed. These areas would require rigorous check-scaling at appropriate intervals. Check-scaling should be conducted after each blast in the ore drives where mesh has not been installed.

16.2.9.6 Geotechnical hazard management

AMC recommends a system be implemented for formally managing the geotechnical hazards associated with each underground mine. This would involve defining how the measures above are implemented. Geotechnical hazard management would consist of three main elements:

- A ground control management plan (GCMP), which includes but is not limited to the following:
  - Description of the mine design basis.
  - Description of specific geotechnical hazards and how they are managed.
  - Monitoring and data collection requirements.
  - Processes for integrating new data into the mine design.
  - Processes for reporting geotechnical hazards / incidents.

  - Responsibilities of key mine personnel, such as the mine manager, shift supervisors, geologists, geotechnical engineers, surveyors and workforce in general.
  - Development of ground support standards based on expected rock mass conditions on implementation, as part of a ground control management plan.
- A geotechnical hazard register, which includes all current geotechnical hazards including development and stope hazards, the status of each hazard, and the strategy for risk management of each hazard.
- Trigger-action-response plans (TARPs) for managing geotechnical hazards. The TARPs would identify the required actions and responsible personnel for geotechnical hazards of varying degrees of potential risk.

16.2.9.7 Backfill requirements

Backfill will be critical to ensure that large stoping areas are not left unfilled for extended periods, reducing confinement and allowing pillars to shed load into adjacent excavations, should extraction ratios exceed 60%. The inclusion of barrier pillars between stoping blocks will assist to reduce stress redistribution that can concentrate loads on slender pillars. In future, primary drifts size should be limited on excavation size and include cablebolt reinforcement.

OreWin (2023) calculated the required backfill strength to be 130 kPa to achieve a free-standing height of 7.0 m. In areas with drift and fill, it is expected that no primary drifts have paste wall exposure. The central secondary between both primaries is expected to have paste exposure side walls predominantly 5 m to 7 m height.

Backfill strength in single cut primary drifts is recommended at 500 kPa, and best practice tight filling by area must be achieved. The strength requirements may be reviewed following:

- Consistent in situ strength testing, particle size distribution and the overall rheology of the mix monitoring.
- Indications that monitoring of backfill exposures are performing well, not exposing personnel to spalling.
- Underground closure measurements regularly carried out in primary drifts.

QA/QC control on the backfill properties should be implemented at the backfill plant. This includes controlling water cement ratios, producing a backfill mix with consistent slump, which is also important to minimize the increase of friction inside the pipes.

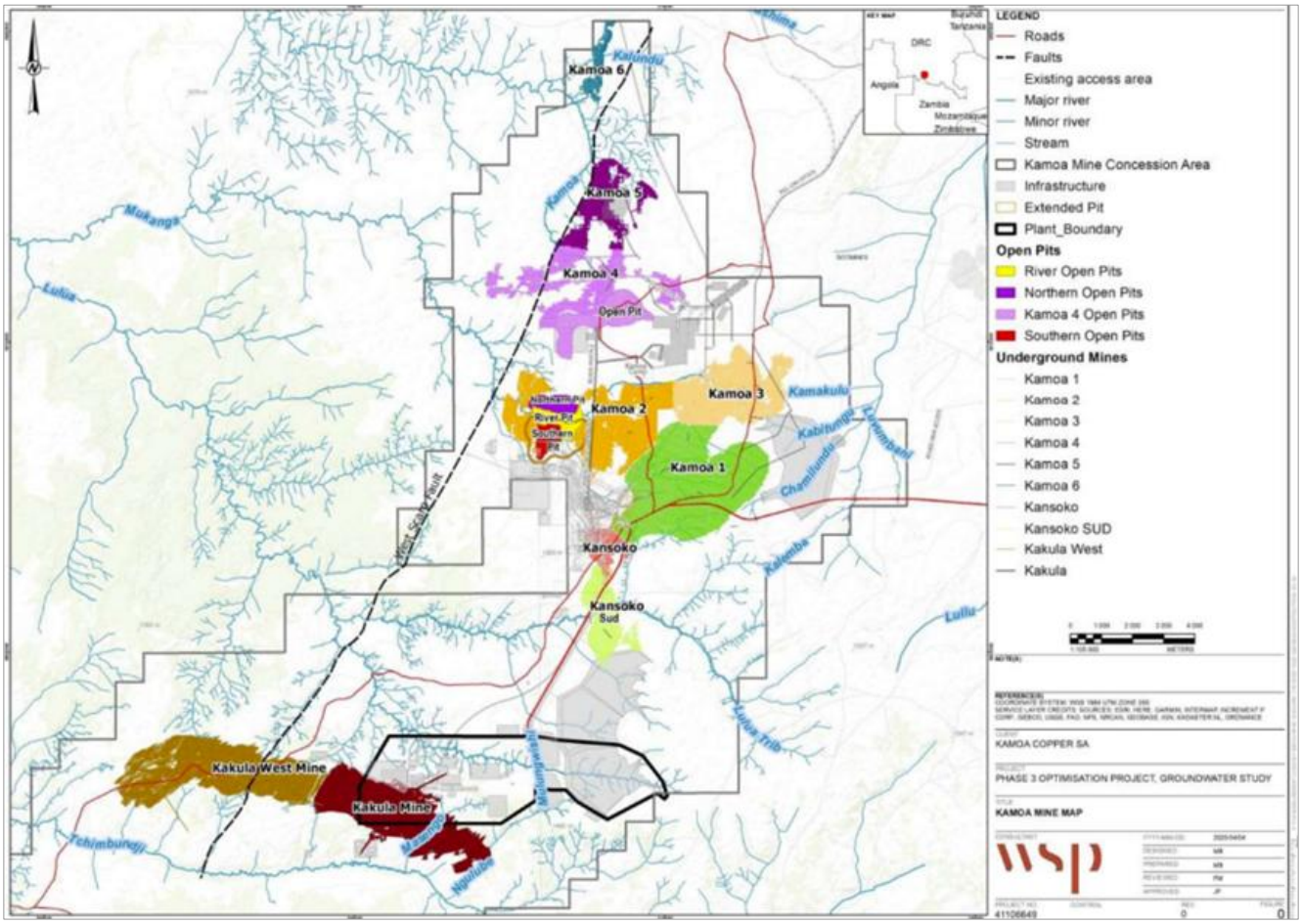
16.2.10 Hydrogeological setting

Hydrogeological studies indicate where broken ground is intersected, these will likely form conduits for water. Figure 16.42 shows the Kamoa mine locations in relation to surface water drainage. The key water bearing feature is the rechargeable West Skarp fault that dissects Kakula West and extends north to the western side of Kamoa 4 and 5 and eastern side of Kamoa 6.

The major aquifer in Kakula and Kakula West is the Roan Sandstone (R4.2 unit) in the footwall that is recharged from surface and is approximately 1,000 m thick. In the hangingwall, the Upper Diamictite includes a series or splay faults that bring water to the mining zones. Mining must avoid mining near the West Skarp fault or into the footwall sandstone until dewatering. Permeability of the Upper Diamictite is relatively well developed and groundwater inflow requires active management during mining.

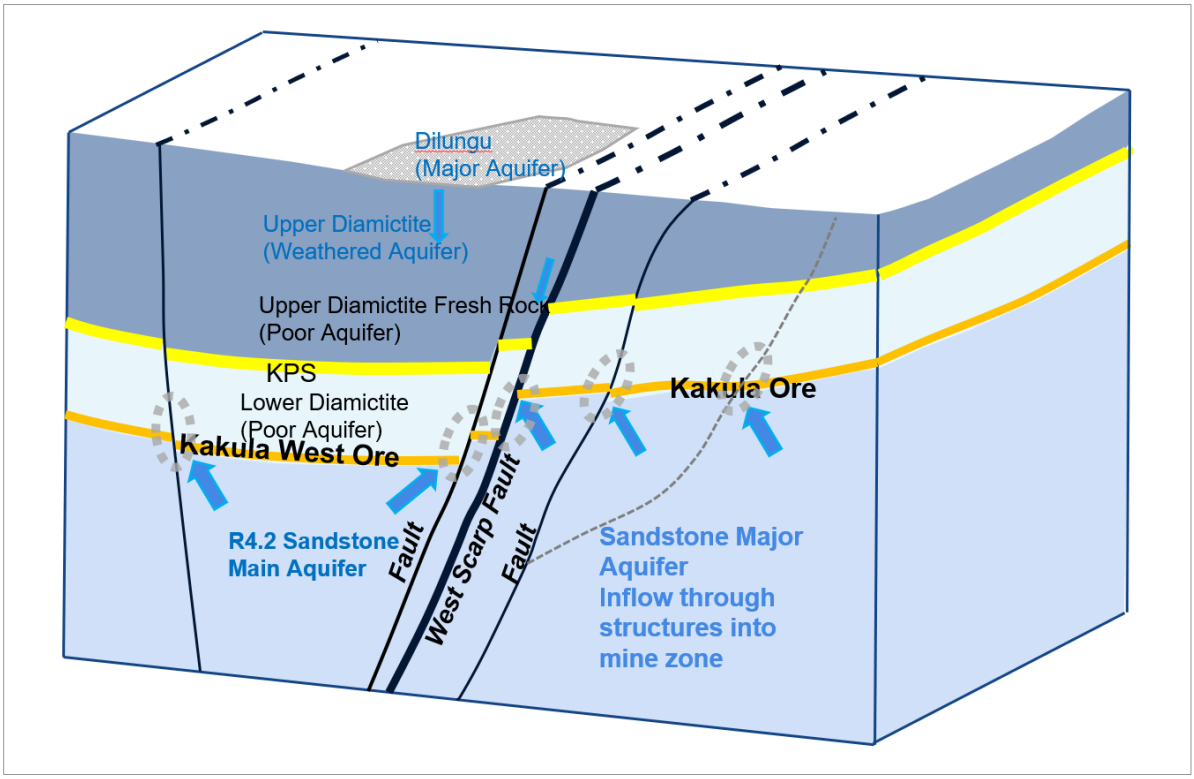
In Kakula, water intersections have been shown to form parallel lines to the West Skarp fault related to splay faults as shown in the simple water movement concept in Figure 16.43.

Figure 16.42 Kamoa mine areas with surface water drainage



Source: WSP, 2025

Figure 16.43 Schematic- Kakula – Kakula West simple water movement concept



Source: WSP 2025

16.2.11 Historical production

KCSA is mining and operating the Kakula, Kansoko, Sud, Kamoa 1 and Kamoa 2 Mines. Operations started during April 2020 with the mining of the Kansoko Mine and later the Kakula Mine which were amongst the first operations being brought into production.

Kamoa-Kakula development plan and production history includes:

- Stantec and KGHM Cuprum (2019) Kamoa 2019 PFS and Kakula PEA. Underground mine planning and design. Included 7 m high cut and fill lifts, with multiple planned to achieve orebody thickness.
- OreWin (2019) –Kakula 2019 PEA analyses, Kakula and Kamoa North underground mining, Kamoa-Kakula combined production schedules, and financial models. Overall report preparation. It is understood that the schedule was not timed in multiple lifts however full height ore was extracted. The mining method experienced instability due to slips along the exposed long openings. The mining method was changed to room and pillar in these circumstances.
- OreWin (2020) - Kamoa-Kakula Integrated Development Plan.
- OreWin (2023) - Kamoa-Kakula Integrated Development Plan.

16.2.12 Life-of-mine context

Kakula Mine experienced a significant seismic event on 18 May 2025, resulting in damage to critical excavations and infrastructure, including damage to the dewatering system resulting in flooding, and with all personnel safely evacuated. Surface InSAR satellite-based remote sensing verified that a substantial

surface area had shown subsidence indicating slip and failure of the underground supporting pillars. A number of factors have been identified and lessons learned for the context of future mine design:

- The event has been attributed to cascading failure through parts of the inner mining precincts, exacerbated by major geological structures and high extraction ratios. Several design features of the mine, such as limited stabilization pillars, wide block spans, and high initial extraction ratios, contributed to the instability.
- The outer precincts also experienced significant damage, in a pattern that indicates excessive initial extraction weakened those areas.
- Mine design and schedules now include a more regular system of stabilization pillars, a lower extraction ratio during development and excessive early development, and an extraction sequence that strategically manages load and deformation based on numerical modelling.
- Kakula mine is currently in a stabilization phase, focusing on safe recovery of the operation. Immediate tasks include designing a new pillar system, confirming the best mining methods for each mining area, confirming safe accessways, and developing an extraction strategy that stages recovery to preserve accesses, establish contingencies and mitigate stability risks. Learnings from Kakula Mine will be applied across all deposits.
- Kakula mine will re-establish following a modified plan that incorporates lessons from the event and reduced recovery but will be heavily dependent on new mining areas to re-attain a high level of reliable production.

16.2.12.1 Mine planning guidance

Guidance on dimensions for regional planning layouts was provided by Beck Engineering (2025) as a result of the review and numerical modelling of the Kakula mine failure. Table 16.31 lists the mine planning design dimensions that have guided the layouts for each mine at pre-feasibility design phase.

Table 16.31 Mine planning guidelines (adapted Beck Engineering, 2025)

Element	Guidance:
Block spans (blocks divided into panels)	< 300 m x < 300 m for a group of 2 to 4 x 2 to 4 panels, unless analysis permitting a larger block is confirmed by the geotechnical superintendent and Mine Manager. Most blocks will be smaller accounting for geological complexity.
Panel Widths	80-110 mW and 80-110 m Length. targeting 3 panels wide as much as possible.
Panel Length	Not more than ~60-120 m, with up to 3 end to end, separated by chain pillars 2 to 2.5 W:H chain pillars. Which should consider conditions and major structures, folds etc. Generally, total length of panels end to end + pillars should target < 300 m.
Inter block pillars (Access not critical after blocks are complete)	6 x 6 m drifts preferred, 21 m skin to skin 7 x 7 m drifts if needed, 25 m skin to skin 48 m wide when finished Recovery on retreat if safe to do so when no longer required.
Inter block pillars (Long term important accesses)	6 x 6 m drifts preferred, 21m skin to skin 7 x 7 m drifts if needed, 25 m skin to skin 75 m wide Recovery on retreat if safe to do so when no longer required.
In panel crosscut spacing	6 x 6 m drifts preferred, 21 m skin to skin 7 x 7 m drifts if needed, 25 m skin to skin
In panel, sub stope scale rib and chain pillars	Spaced to achieve approximately 75% recovery in each panel but assume a contingency to manage stope problems or adverse geology.

Element	Guidance:
Contingency pillars	Brackets around all major pillars will be maintained as contingency pillars and only extracted if performance indicates that is acceptable. i.e. final widths above are after the contingency pillars are recovered.
Panels mined primary-secondary or primary only, subject to conditions	Extraction of secondaries, and degree or secondary extraction planned by sufficient analysis, but subject to re-assessment based on observation, measurement and analysis during the stoping phase.
Decision points	Kakula will employ an observational approach involving staging access to new areas to gain orebody knowledge, and monitoring and measurements during operations to: - test and update the empirical tools when needed, to audit and improve their performance. - manage variability and rock mass defects found during development. - confirm the design assumptions. - identify emerging hazards. - confirm that the design control measures are being implemented and are effective. - before a design control measure is 'locked in' (the sequence within a block is locked in, stopes near critical infrastructure are mined, a stope is mined that sets the pillar span, or the turnout for a sill or crown level is mined for example), that the stability performance still supports the design in that area, and that the latest geotechnical information has been considered. At each stage of extraction, the decision to proceed will be based on an interpretation of whole of system response to confirm the extraction can proceed.
Primary or secondary stope advance distance before backfilling	Maximum 2.5 x the stope width
Bracket pillar	Kakula West – East and West, a 20 m pillar has been left on either side of the faults to account for anticipated poor ground and challenging recovery.

Pillar type definitions:

- Barrier / Contingency pillar:
  - No stoping, minimal development, permanent.
  - Intent: to prevent the propagation of a failure to adjacent work areas.
  - A backfilled stope is not a pillar.
- Access corridor / bracket pillar / inter block pillar (long term):
  - Between zones.
  - No stoping, extracted on retreat when no longer required, if conditions permit.
  - Intent: to preserve safe access and prevent propagation between blocks.
  - A backfilled stope is not a pillar.
- Stabilization pillar / temporary access pillar / inter block pillar (short term):
  - A semi-permanent or permanent pillar inside a stoping zone to reduce deformation in the block workings, or to manage the impacts of a geological defect.
  - The stabilization pillars limit failure inside a block, if it occurs.
  - A budget for stabilization pillars in each block is assumed, but they are deployed tactically and are usually used as access within the block.
- Rib / Chain / island pillars:
  - A system of pillars within a stoping block or room to maintain local stability.
  - These pillars are expected to yield. Aim is to reduce deformation in the stoping front or adjacent temporary access only while local work activities require them.

16.3 Mining district hydrogeological assessment

16.3.1 Dewatering system

Kamoa / Kakula dewatering designs can be summarized 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.

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 in turn feed 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.
- **Vertical dewatering 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 2,000 l/s.

Specific dewatering systems and groundwater ingress estimates over the life of each operation are described in the individual deposit in Section 16 and Section 18 below.

16.4 Life-of-mine summary

16.4.1 Life-of-mine strategy selection

The strategy applied to the KCSA deposits is to maximize the value generated from the established infrastructure and installed capacity of the concentrators and smelter in place at KCSA. The production profile for the KCSA deposits for the life of the operation is planned to maintain operation of these facilities at full capacity of 17 Mtpa milled ore.

At the time of reporting, a strategy optimization process using a Hill of Value™ approach has been initiated. This process will enable a dynamic approach to cut-off grade selection, mining rates and sequence of mining in line with the development of the multiple deposits of the KCSA operations.

16.4.2 Cut-off grade approach

To date, KCSA has relied on a break-even cut-off grade approach to differentiate between full grade ore, marginal ore and waste. Previous studies have applied a Net Smelter Return (NSR) value as the basis of determining cut-off values. This approach has not been applied to the IDP in this 2026 report. Due to the only material revenue generating component being copper, total copper grade (%TCu) has been used as the cut-off value determinant.

For the development of this IDP 2026 mining plan, rather than apply a uniform cut-off across all deposits, KCSA has evaluated life-of-mine plans for each deposit considering a cut-off grade of between 2.0%TCu and 1.5%TCu. Mine plans for each deposit were developed and the differential economics of either scenario was compared to determine a cut-off approach. The varying characteristics of each deposit, and

the intention of maintaining reliable mining parameters and geotechnical controls has resulted in each scenario applying both an economic cut-off, practical mining parameters and geometric considerations to differentiate between ore and waste.

16.4.3 Mining method description

KCSA deploys several underground mining methods across the various deposit configurations in the currently operating mines. Additional mining methods have been developed for future underground mining where the deposit geometry requires.

The active and planned underground mines exploit stratiform orebodies with dips ranging from approximately 9° to 35°, averaging 18°. Orebody thickness varies between 2.5 m and 20 m, with a typical average of 6 m. The deposits occur across a range of depths, from surface outcrop to 1,400 m below ground level (mBGL).

This deposit disposition dictates and limits the available choice of mining method. The stratiform, flat dipping nature of these narrow deposits precludes typical longhole mining methods such as sub-level and longhole open stoping. The mining methods found to be best suited to these deposits are variations on drifting methods. These methods are outlined in the table below in Table 16.32.

Table 16.32 Underground Mining methods applied at KCSA

Mining method	Application at KCSA
Drift and fill (D&F)	Flat to shallow dipping (0 to 20 degrees), deposit thickness from 2.5 to 20 m
Cut and fill (C&F)	Moderate dipping (30 to 40 degrees), mining thickness 5 to 15 m
Room and pillar (R&P)	Flat dipping (0 to 5 degrees) with mining heights below 6 m
Long-hole Stoping (LHS)	Steep dipping (50 to 90 degrees), mining thickness 4 to 10 m

Several of the deposits outcrop or are near surface and could be considered as potentially open pit operations. The narrow nature of the orebody would result in a very high stripping ratio with waste to ore ratios quickly reaching 15 waste tonnes generated for every ore tonne recovered. KCSA has maintained a strategic intent to mine these deposits through underground methods, limiting surface disturbance and the generation of large waste dumps.

16.4.3.1 Drift and Fill mining method

The most prolific mining method deployed in the KCSA underground mining operation is Drift and fill mining.

The drift and fill mining method is applied as described in the following sequence:

Access development

Access drives are developed along the apparent dip at gradients selected to ensure efficient tramming of load-haul-dump (LHD) equipment.

Ore drift development

Ore drifts are developed between two access drives at maximum dimensions of 6 m (width) by 6 m (height). Ground support is installed progressively during development and includes systematic cable bolting at regular intervals.

Retreat mining

Upon breakthrough of the ore drift to the adjacent access drive, retreat mining is initiated. The retreat sequence involves longhole stripping of the drift walls to a maximum depth of 7.5 m on both the left and right sides, with additional upward stripping where mineralization extends above the drift horizon. The retreated zones constitute unsupported, non-entry areas; accordingly, all loading operations within these stopes are conducted using remote-controlled LHD equipment.

Backfilling

Backfill barricades are constructed and the resulting void is filled with cemented fill either upon completion of the full retreat sequence or at pre-determined positions along the retreat horizon, as dictated by the mine plan.

Primary and secondary drift sequence

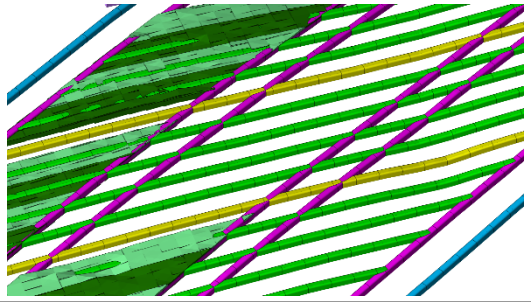
Ore drifts are extracted in a primary and secondary sequence. Primary drifts are mined, retreated, and backfilled first. Secondary drifts are developed in sequence with primary drifts; Retreat mining begins after the backfill in the primary drifts has cured. During secondary drift retreat, the longhole stripping will expose the cemented backfill of adjacent primary drifts. This sequencing ensures that backfill walls are only exposed within the non-entry zone, eliminating any direct personnel exposure to backfill faces.

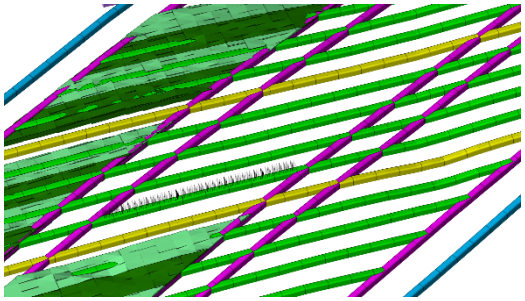
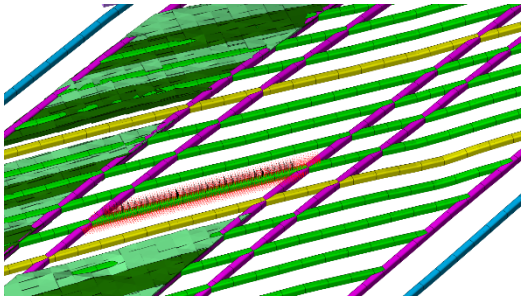
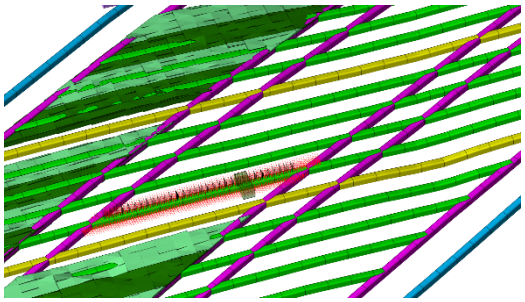
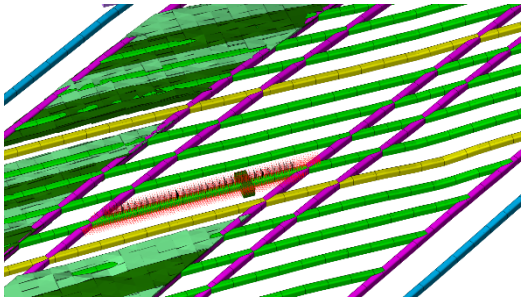
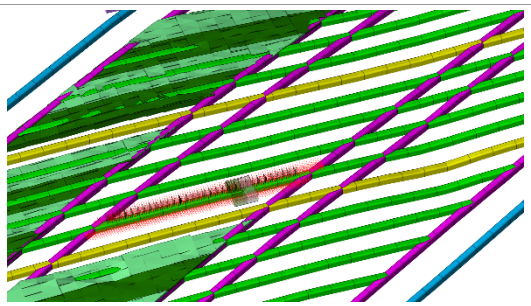
Pillar recovery

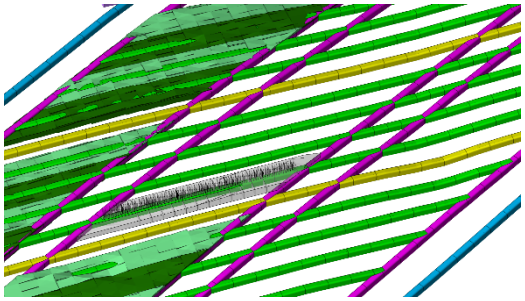
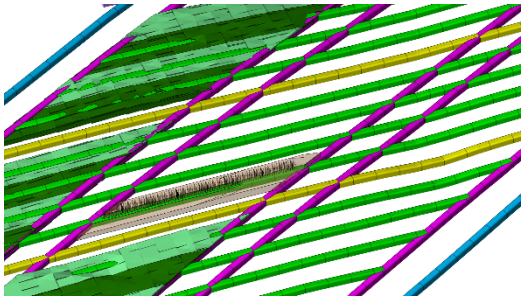
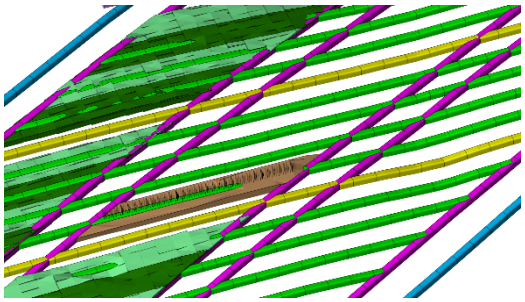
Local and regional pillars are incorporated into the mine design and are maintained in place throughout the extraction of each mining block. Upon completion of primary and secondary mining within a block, selected pillars will be subject to partial extraction on a retreat basis. Pillar recovery will be governed by rigorous geotechnical modelling and ongoing observational analysis. For the purposes of mine planning, a maximum pillar extraction rate of 30% of total pillar inventory is assumed at the end of the mine life.

The Drift and Fill mining sequence is illustrated in Table 16.33.

Table 16.33 Drift and Fill mining sequence

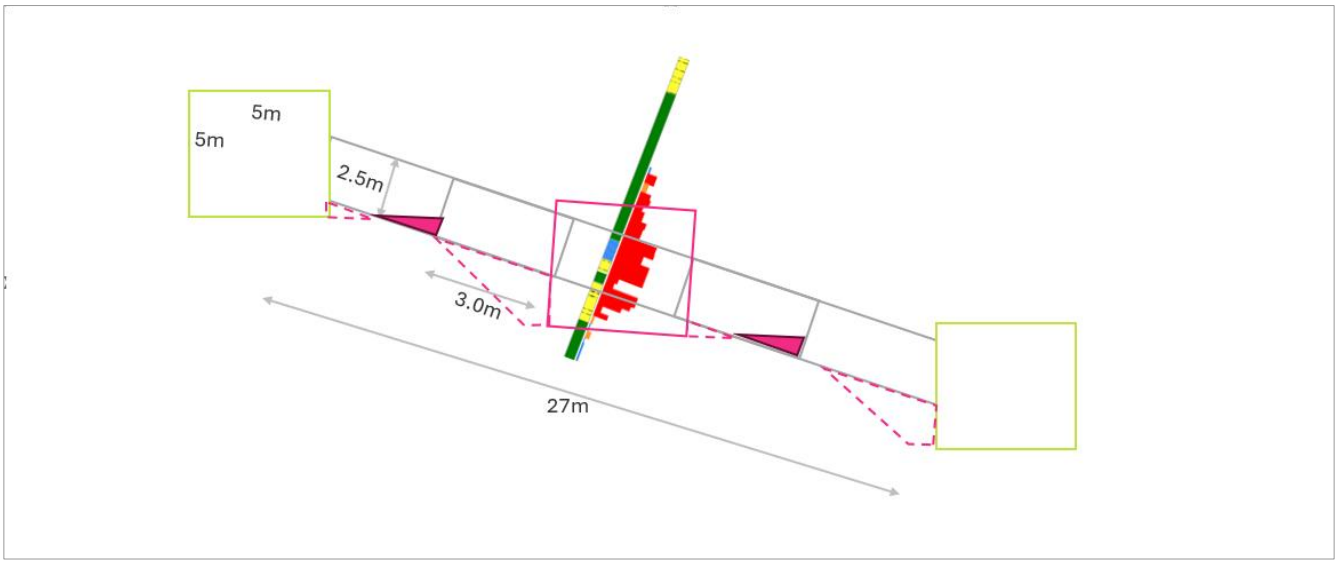
Step #	Step	Description
1		Development. Surrounding development is completed to ensure production and development activities do not overlap and minimize congestion.

Step #	Step	Description
2		Cable Bolting. Cable bolts are installed within the stope footprint to support stope backs for the duration of the stope life, enabling backfilling upon completion.
3		Production Drilling. Production drilling is conducted prior to the stope being commenced to reduce interaction and stope standing times. Variation may be necessary if local poor ground conditions are experienced.
4		Production Firing. The initial slot firing is mined. Multiple options exist for the initial void creation. These will be assessed on a case by case basis, depending on stope thickness, stope gradient, surrounding access etc.
5		Production loading. Conventional bogging will be limited due to the shallow stope thickness. Bogging will largely occur via remote loader.
6		Stope Cycle. The stope will then progress through charging, firing, bogging and recon cycles until its completion.

Step #	Step	Description
7		Stope Closeout and Backfilling Setup. Upon completion the stope closeout process will commence, including reconciliation. backfill walls will be constructed at accesses and filling lines installed.
8		Backfilling. Backfilling will occur in one or more passes depending on stope gradients. Adjacent drives will be impacted through exclusion zones. Some drives may be isolated due providing access for filling.
9		Backfill Curing. Backfill curing is allocated to take 30 days prior to unlocking the adjacent stope.

The extraction profile for a typical drift and fill layout in cross section is illustrated in Figure 16.44. Typical planned dilution and ore loss is also indicated. The mining width ranges between 2.5 m and 20 m and the dip ranges between 0° and 20° across the various deposits. As the thickness and dip changes, the dilution and mining recovery will change in proportion, to fall within the ranges indicated elsewhere in this document.

Figure 16.44 Drift and fill cross section with indicative drill hole and mineralization.



Source: AMC, 2026
Note: Illustrative drift and fill design with longhole extraction between the drifts. Minimum height of 2.5m shown. Indicative drill-hole with mineralization shown and the red profile indicates copper grade.

16.4.3.2 Cut and Fill mining method

Cut and Fill mining is a modified stoping method using wall slashing and cut-and-fill with cemented fill as shown in Figure 16.45. Figure 16.46 represents the sequence of mining for the cut-and-fill accesses for each level. During development of the decline, close spaced drilling to confirm the location of the ore contacts will enable detailed design of the gradients for each access crosscut. The executable design should include alternating level accesses offset to avoid the creation of any thin floor pillars.

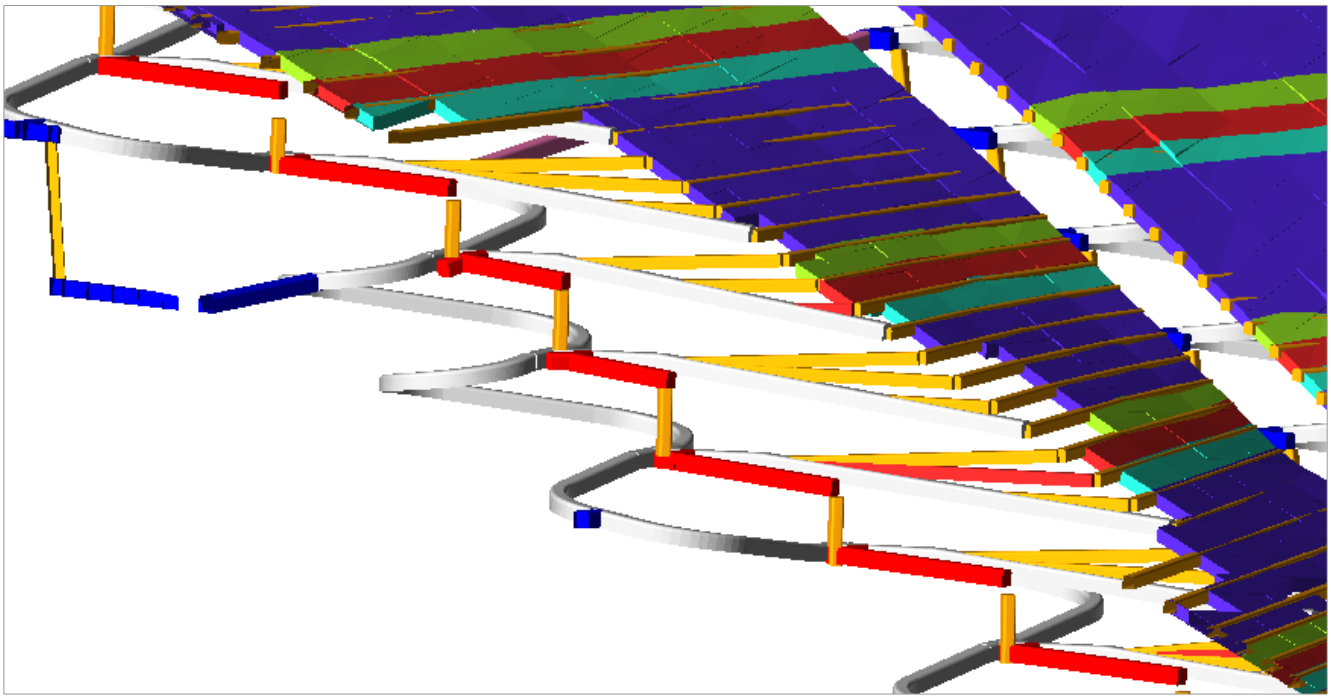
The mining sequence proceeds from the access and along strike via an ore drive as shown in Figure 16.47 A. A stope is excavated up dip from the ore drive to allow blasting rings into a free face. The shallow dipping ore will require a bogging/loading horizon to be excavated along with the stope, which will allow for material to rill (ie. Arrive) at the draw point for loading. This causes up to 17% external dilution for ore thickness of 4.5 m, but down to 0% external dilution for ore thicker than 8.5 m.

Each 40m strike sidewall stope will be paste filled via two short upholes from near the brow. One hole for backfilling and one for allowing air escape for tight fill. These are shown in blue near the brow in Figure 16.47. Once the entire level is paste filled, the next cut-and-fill access and ore drive is mined along the upper paste contact and the process repeats.

The design includes 50 m wide regional pillars spaced at 400 m, with 40 m wide access pillars in between which are sometimes recoverable on retreat when they remain accessible. In most cases access is lost due to the cut-and-fill access method and the access pillars remain in situ.

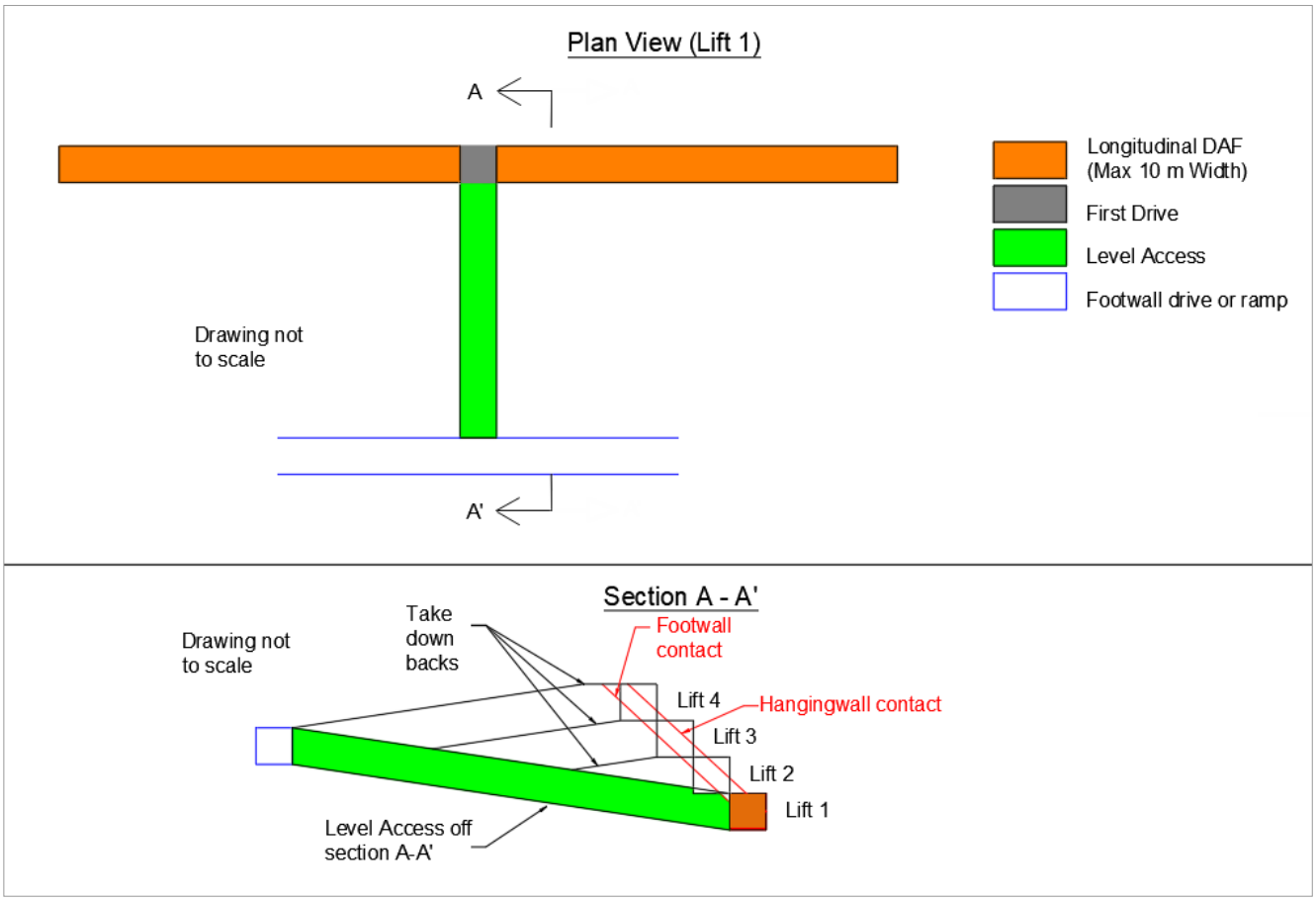
The production blocks, delineated in purple in Figure 16.78 are isolated by sill pillars, coloured in red. These red sill pillars may only be recovered once all other activities are completed above it. Most are recovered at the end stage of the mine life. Figure 16.77 and Figure 16.79 show isometric views of Kamoa 3 with stopes removed.

Figure 16.45 Conceptual cut-and-fill access



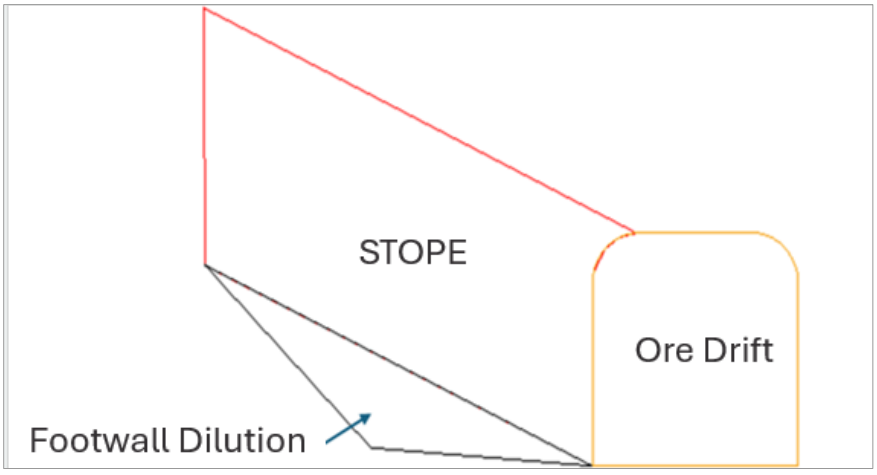
Source: AMC 2026

Figure 16.46 Drift and Fill extraction sequence



Source: AMC 2026

Figure 16.47 Cut and Fill extraction



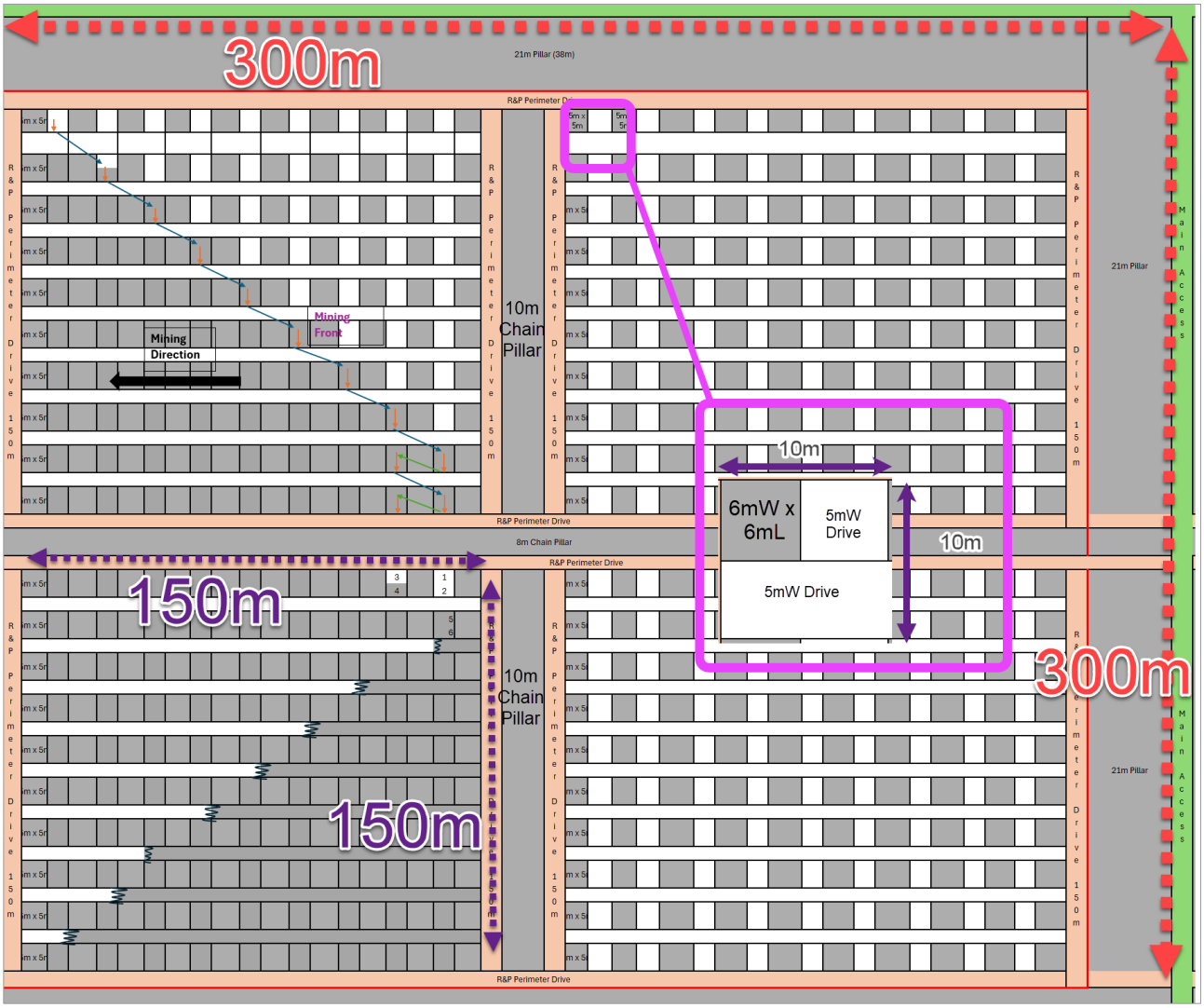
Source: AMC 2026

16.4.3.3 Room and Pillar mining method

For room and pillar mining (R&P), production is organized into nominal production blocks of approximately 300 m × 300 m equivalent area, noting that block geometry is adjusted as required to suit local orebody geometry, access constraints, and infrastructure alignment rather than being strictly square. Each production block is subdivided into four panels of approximately 150 m × 150 m equivalent area, separated by 10 m-wide chain pillars. Most panels are designed to operate as independent mining fronts, providing flexibility in sequencing and supporting multiple concurrent production fronts.

Mining commences with the development of perimeter drives to production panels and blocks and establish ventilation and services. Following completion of the perimeter drives, panel entry drives are developed from east to west to open individual panels, after which crosscuts are developed from north to south in a single direction, with a lagged mining front retreating back toward the main access.

Figure 16.48 R&P layout and planned sequence



Source: AMC 2026

Within a typical R&P panel, recovery is constrained by the requirement to leave regularly spaced pillars to maintain stability. For the illustrated layout, a panel module of approximately 10 m × 10 m comprises two 5 m wide drives separated by a 6 m × 6 m pillar, resulting in a panel-scale extraction ratio of approximately 75%.

The R&P layout adopts a pillar geometry with a height-to-width ratio of 1:1.2, selected to provide stable excavation conditions within the thin orebody. When allowing for expected overbreak, drive dimensions have been standardized to 6 m wide × 6 m long, providing local inter-panel stability and treated as non-recoverable.

Chain pillar dimensions are governed by the extraction drive height. In accordance with Beck Engineering guidance, a 1:2 height-to-width ratio is applied, resulting in chain pillars of appropriate width to provide regional stability between production panels. Regional pillars are scheduled for extraction toward the end of the production life, subject to prevailing ground conditions and operational constraints, with the mine plan assuming an overall 30% extraction ratio. Further detail on pillar design rationale and recovery assumptions is provided in the Beck Engineering guidance.

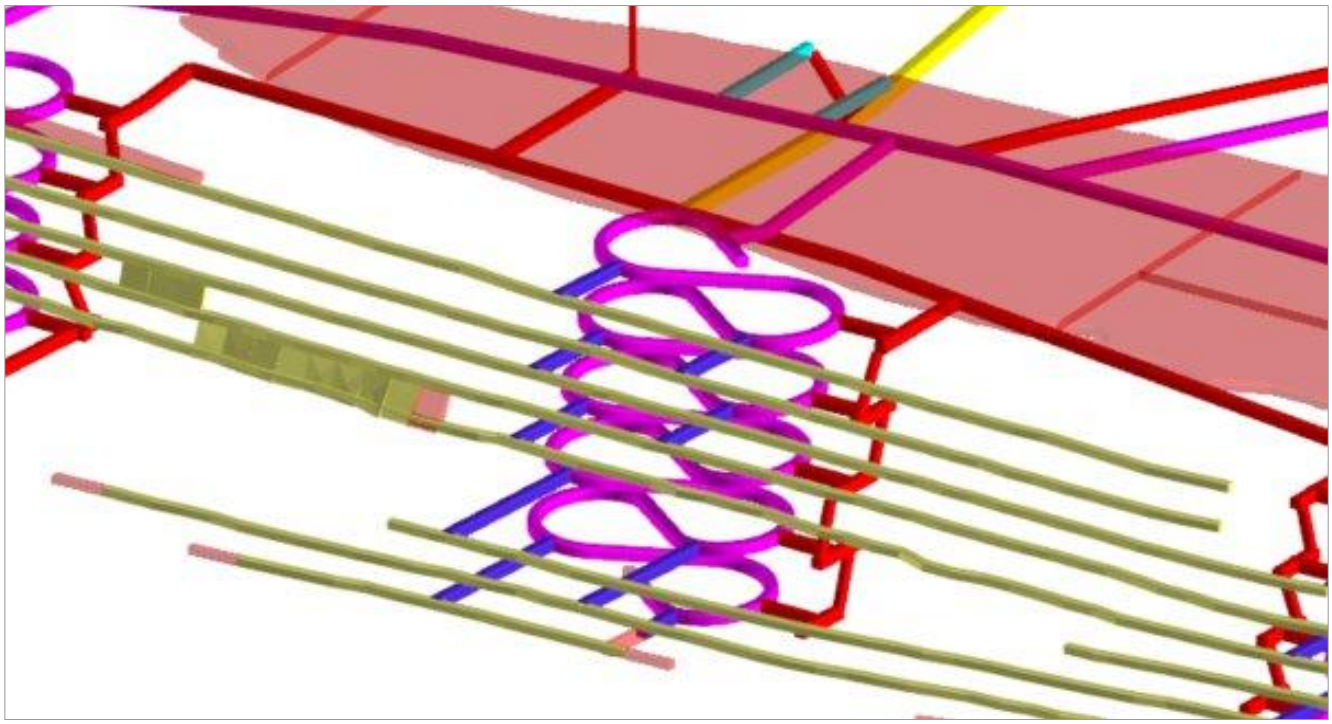
16.4.3.4 Longhole stoping mining method

Longhole stoping (LHS) is the selected mining method for domains where the orebody dip exceeds 55°. LHS is classified as a bulk mining method and offers a high degree of operational efficiency through its suitability for mechanization and the productivity gains associated with large-scale drill-and-blast extraction.

A decline is developed to the lowest elevation of the designated stoping area. Level accesses are established from the decline at each production level, from which longitudinal ore drives are developed to define the lateral extent of the stoping area.

In ground classified as fair to poor quality, a conservative stope design philosophy has been adopted. Level spacings and individual stope lengths are limited to 15 m, reflecting the reduced stand-up capability of the host rock mass and the need to manage dilution and geotechnical risk within acceptable limits (see Figure 16.49).

Figure 16.49 Long hole stoping schematic



Source: AMC 2026

16.4.4 Mine ventilation and cooling

16.4.4.1 Summary

The ventilation and cooling designs are robust, technically sound, and aligned with the mine's LOM production schedule. The system accommodates the increased demands of deeper mining by integrating phased infrastructure upgrades, validated modelling, and best practice approaches. Key components, including the staged implementation of main fans on surface, optimized ventilation flows, and a cooling strategy using surface BACs, ensure compliance with Kamoa SA design criteria, notably maintaining workplace temperatures below 27.5°C WB and diluting diesel exhaust gas and particulate matter.

A series of interactive ventilation and thermal network models were developed to simulate defined LOM "snapshot" years, representing progressive increases in mining depth, production, and diesel equipment. These simulations quantified the required airflow quantities, main fan duties, refrigeration capacity, and electrical power absorbed to maintain compliance with the design limit of 27.5°C wet-bulb temperature in work areas. Airflow requirements were determined using first principles and were guided by the most demanding of diesel dilution, heat dilution, or minimum velocity criteria.

Critical factors used in determining ventilation and cooling requirements included a summer ambient surface air intake temperature of 22.0/30.0°C wet-bulb / dry-bulb (wb/db), heat flow from surrounding rock with a maximum virgin rock temperature (VRT) of 43.5°C in Kamoa 1 mine, dilution of diesel exhaust gas (0.063 m^3/s per rated kW at the point-of-use), and minimum air velocities.

The auto-compression will create the most significant heat source accounting for 43% of the mine heat. Auto-compression diesel equipment will account for 25%, and heat flow from the rock will be 9%. The

remaining heat (23%) will come from secondary fans, conveyors, and other sources, including electrical equipment, pumps, etc. The overriding function in determining the air quantity is diesel exhaust dilution.

The mine design and schedule proposed for the 36-month and LOM projects for Kakula, Kamoa 1, Kamoa 2, Kamoa 3, Kamoa 4, Kamoa 5, Kamoa 6, Kansoko Sud and Kakula West projects can be ventilated and cooled to provide suitable workplace conditions over the LOM. BBE collaborated with the ventilation, engineering, and mining teams to ensure the design met the required ventilation volumes and cooling requirements.

Refrigeration is required at Kakula (18 MWc), Kamoa 1 (17 MWc) and Kakula West-West (9MWc) due to the depth below surface and the associated diesel fleet. The refrigeration infrastructure will be introduced incrementally as required, aligning with the mining schedule to optimize capital expenditure and ensuring operational readiness.

16.4.4.2 Design criteria

The project applies the general ventilation criteria listed in Table 16.34.

Table 16.34 Ventilation design criteria

Parameter	Unit	Value
Surface - Summer ambient temperature 90th%	ºC wb/db	22.0/30.0
Surface barometric pressure	kPa	85.4
Underground Workplace temperatures		
Design wet-bulb temperature	°C	27.5
Maximum wet-bulb temperature	°C	32.5
Maximum dry-bulb temperature	°C	37.0
Diesel exhaust dilution	m^3/s per rated kW	0.063
Minimum velocity for active stopes	°C	≥ 0.4 m/s
Intake airway velocity	m/s	≤ 6.5
Return airway velocity	m/s	≤ 8
Upcast shaft velocities	m/s	≤ 22
Downcast shaft velocities	**m/s**	**≤ 12**

Note: Avoid velocities between 7-12 m/s in upcast shafts and raises to avoid water blanketing.

16.4.4.3 Design basis

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

The primary ventilation system will be designed as an exhaust system. Main fans will 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 airflow rate and will consider utilization.

Primary leakage rate used for preliminary airflow estimates for when the mine is fully developed are 10%. These factors are used to maintain appropriate working temperatures and minimum velocities throughout all the openings.

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 activity types for both trucks and LHD's. First-principles calculations were used to determine the heat load from vehicles based on the work output for each activity.

Diesel engines are assumed to have a power conversion efficiency of 35%.

16.4.4.4 Assumptions

The following assumptions are made:

- There are no known abnormally high concentrations of contaminants such as radon, flammable gases, toxic gases, and silica within the strata.
- Sufficient raise bore machines and raise bore strings will be available and can be mobilized promptly to develop the significant number of raises required initially.
- Regarding raise boring, there are no known geotechnical limits, and it is assumed that ground conditions will be suitable for boring shafts up to 6.0 meters in diameter to the surface without the need for pre-sinks.
- Sufficient power supply, water supply, and associated reticulation networks will be available to support the primary ventilation strategy.
- Due to the lack of site-specific information, key heat criteria, including ambient conditions and rock thermal properties, are based on assumed values from BBE's database. The mine is currently undertaking geothermal measurements and once a sufficient database is established, these values will be used in later work.
- Trucks will not enter fan-ventilated stope access drives but will wait in through-ventilation in perimeter drives.
- Loading in the production areas will be done with mechanical ventilation, as blasted ore will block through-ventilation. Fans will supply at least 22 m^3/s at the point of use to maintain effective diesel exhaust dilution.
- Fans will be moved between active working places; 30% additional fans will be allocated to temporarily stopped working places.

16.4.4.5 Other mining assumptions

- Backfill will be mixed on surface and placed in mined-out production drives.
- Fixed-time end-of-shift blasting will take place.
- Aquifers in hang-wall, sandstone fissure water, and other fissure water will be continuously isolated and, if intersected, will be managed by dewatering and will have a minor effect on ventilation needs initially. Ventilation modelling assumes a 20% factor for unconfined water. Later in the LOM, the inflow of water in Kamoa 1 and Kakula increases significantly, and at this point, fissure water at Kakula becomes a heat source.

16.4.4.6 Heat load and ventilation analysis

A heat load analysis was done for each mine to ensure the average 27.5°C WB design temperature is not exceeded. The design process involved solving a series of interactive VUMA-network heat flow and ventilation models.

Mining and development operations were simulated in line with the mining schedule. Heat loads were added for the main operating equipment. Models were used to determine:

- Primary air and mechanical cooling are required to achieve an energy balance.
- Main fan and refrigeration plant duty and power consumption over the LOM.
- Infrastructure, ventilation shafts and connections to distribute air.
- Compliance with design criteria, e.g., maximum WB temperature.

Based on the simulation results, an empirical spreadsheet was used to interpolate the estimated air temperatures, flow rates, heat loads, and potential cooling requirements between the 36-month and LOM ventilation models. The objective was to ensure that the 27.5°C wb reject design temperature was not exceeded by determining the amount and location of (any) cooling required to maintain temperatures within this design limit.

16.4.4.7 Heat loads considered

The VUMA-network models accounted for heat load from surrounding rock (strata), broken rock, heat from diesel equipment, service and fissure water, auto-compression of intake air, auxiliary fans, and other user-defined heat (pumps, sub-stations, workshops, lighting, use of explosives, ingress of groundwater, workers' metabolic functions, etc.). The overall heat load can be broken down into the following principal components:

Diesel vehicle engines: For steady-state modelling, the quantity of heat energy released by diesel equipment can be estimated by applying factors that account for thermal efficiency, roadway gradient and other performance criteria including acceleration and work rates based on operational cycles. The efficiency of an internal combustion diesel engine is 30% to 35%. Thus, the energy required from the fuel is 3x that for the rated engine work. Approximately 10% of the fuel energy used to move a vehicle uphill is converted to a change in potential energy, thus not contributing to the ambient heat load. Diesel heat appears as both sensible and latent heat additions due to the addition of moisture in their exhausts.

Host rock induces a heat load on the intake air by conduction of heat from the hot rock interior to the rock surface and into the ventilating air at a lower temperature. If not countered, the air temperature will rise to the rock surface temperature. This heat component increases with depth as both VRT and the extent of exposed rock surface area increase. This heat load is dependent on the excavation age (newly mined rock surfaces transmit heat more rapidly, but this effect reduces as the rock cools with time).

Broken rock results in a heat load due to reasons similar to the host rock. The blasted rock is exposed to the lower temperature of the mine environment inducing a flow of heat energy towards a state of equilibrium with the air temperature. This heat component increases with depth due to the higher VRT.

Auto-compression of downcast air results in an increase in air temperature due to the conversion of potential energy to enthalpy as the air loses elevation. The auto-compression effect results in a significant increase in intake air temperature as a function of the depth to which the air flows.

Auxiliary fans do not perform any thermodynamic work and all electrical energy supplied to the fan motor is converted into heat energy. The magnitude of this component to the broader mine heat load depends on the method used for secondary ventilation control and the number of operational fans.

This heat load is not depth-dependent but is affected by higher air densities at depth and presents itself as a sensible temperature increase wherever the forced airstream is delivered.

Other heat loads include operation of rock drills, hydropower power packs for drilling, conveyor belt drives, pumps, sub-stations, workshops, lighting, use of explosives, ingress of groundwater, workers' metabolic functions, etc.

16.4.5 Primary fan stations

Ventilation models were developed for steady state production phases, and from these, the primary fan duties at each exhaust shaft were determined. However, special provision has been made for the development phase of the mine, where air quantities drawn through the ventilation shafts will be higher than during steady-state operations. Kamoa has opted to use the current 900 kW trifurcated axial flow fan station for all future fan stations. Simulations predict that these fan stations are suitable for the project.

Note: For Kamoa 2, 4 and 5 bifurcated fan stations *(560 kW) were selected as the required airflow is relatively low. This configuration provides sufficient ventilation capacity while reducing capital expenditure compared with larger trifurcation stations at the other mines*.

To ensure consistency and efficiency across the mine, all fan stations will be standardized to the same specification. If the decision is made to install 560 kW bifurcated fans, then every station will use this configuration; alternatively, if the 900 kW trifurcated fans are selected, all stations will follow that standard. This uniformity simplifies critical spares management, reduces maintenance complexity, and streamlines training requirements. With identical fan-motor sets equipped with VSDs, the onsite team can operate and maintain all stations using a single skill set, minimizing duplication and improving reliability during forward moves or system expansions.

16.4.5.1 Secondary ventilation requirements

The development ends will be force ventilated using 1,400 mmØ 110 kW auxiliary fans with 1,400 mmØ flexible ducting placed within 30 m from the working face. This fan duct system will supply adequate air to the face to dilute heat and gases. Where decline and perimeter drives are beyond 600 m, the drive will be ventilated with dual 1,600 mmØ 132 kW auxiliary fan and 1,600 mmØ flexible ducting.

16.4.5.2 Ventilation and cooling conclusion

The ventilation and cooling designs are robust, technically sound, and aligned with the mine's LOM production schedule. The system accommodates the increased demands of deeper mining by integrating phased infrastructure upgrades, validated modelling, and best practice approaches. Key components, including the staged implementation of main fans on surface, optimized ventilation flows, and a cooling strategy using surface BACs, ensure compliance with Kamoa SA design criteria, notably maintaining workplace temperatures below 27.5°C WB and diluting diesel exhaust gas and particulate matter.

16.5 Kakula

16.5.1 Summary of relevant information

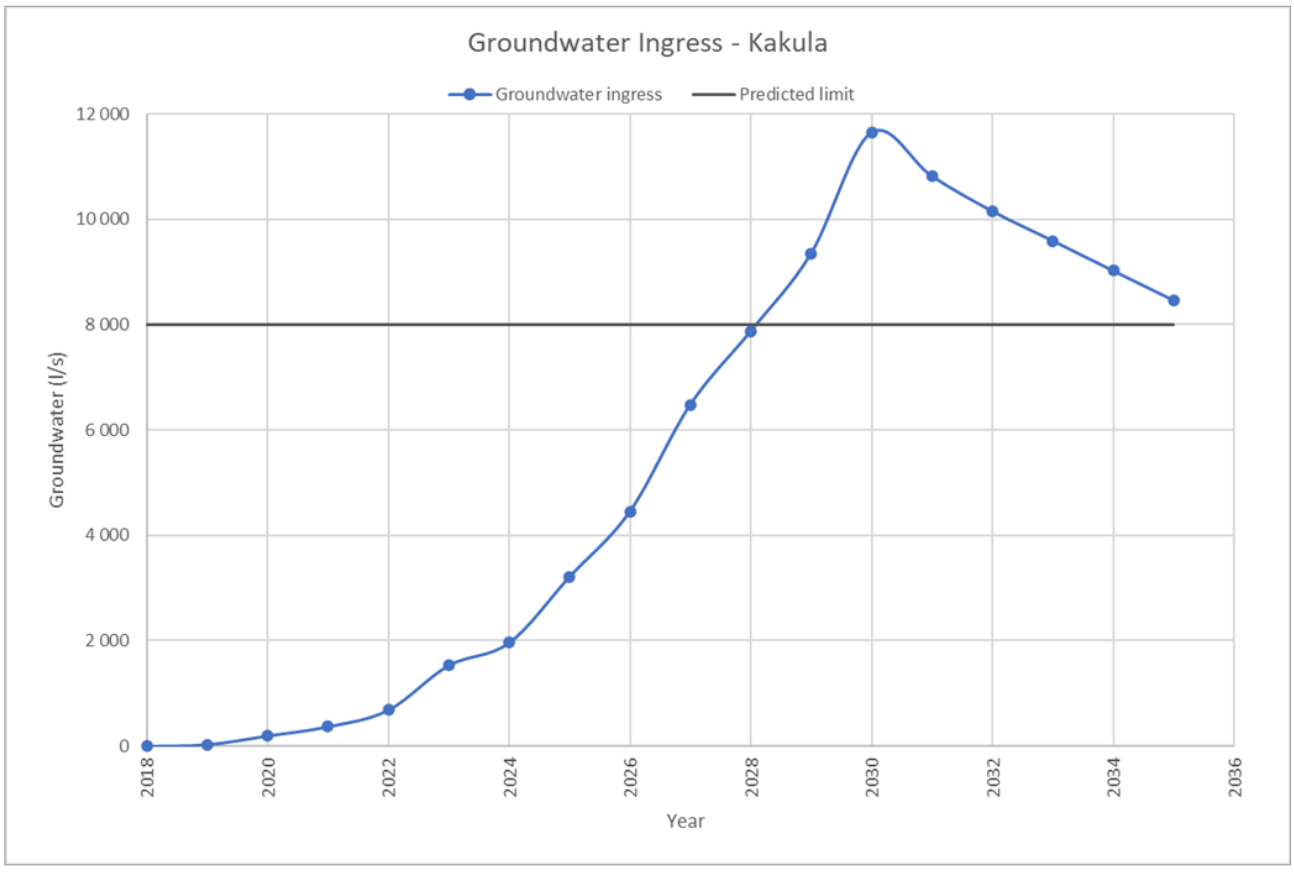
Construction and initial development activities of the Kakula mine were complete in late 2023 and production from this mine has been underway since then. In May 2025 a significant geotechnical event occurred which resulted in significant damage to underground excavations and flooding of a significant portion of the operating areas. (See Section 16.2.12.) Subsequently, significant geotechnical analysis has been undertaken, and this has resulted in a substantial adjustment of mine design parameters, mining approach and extraction sequence. This planning guidance has been applied to all future mine designs for Kakula and for the deposits in the Kamoa-Kakula complex.

As of early 2026, dewatering and rehabilitation activities continue and progress is regularly reported by the company. The Mineral Reserve estimates for Kakula are estimated for the unaffected remainder of the Kakula deposit and future inclusion of affected areas will be dependent on detailed analysis and planning to demonstrate safe and stable mining is achievable.

16.5.2 Dewatering for Kakula

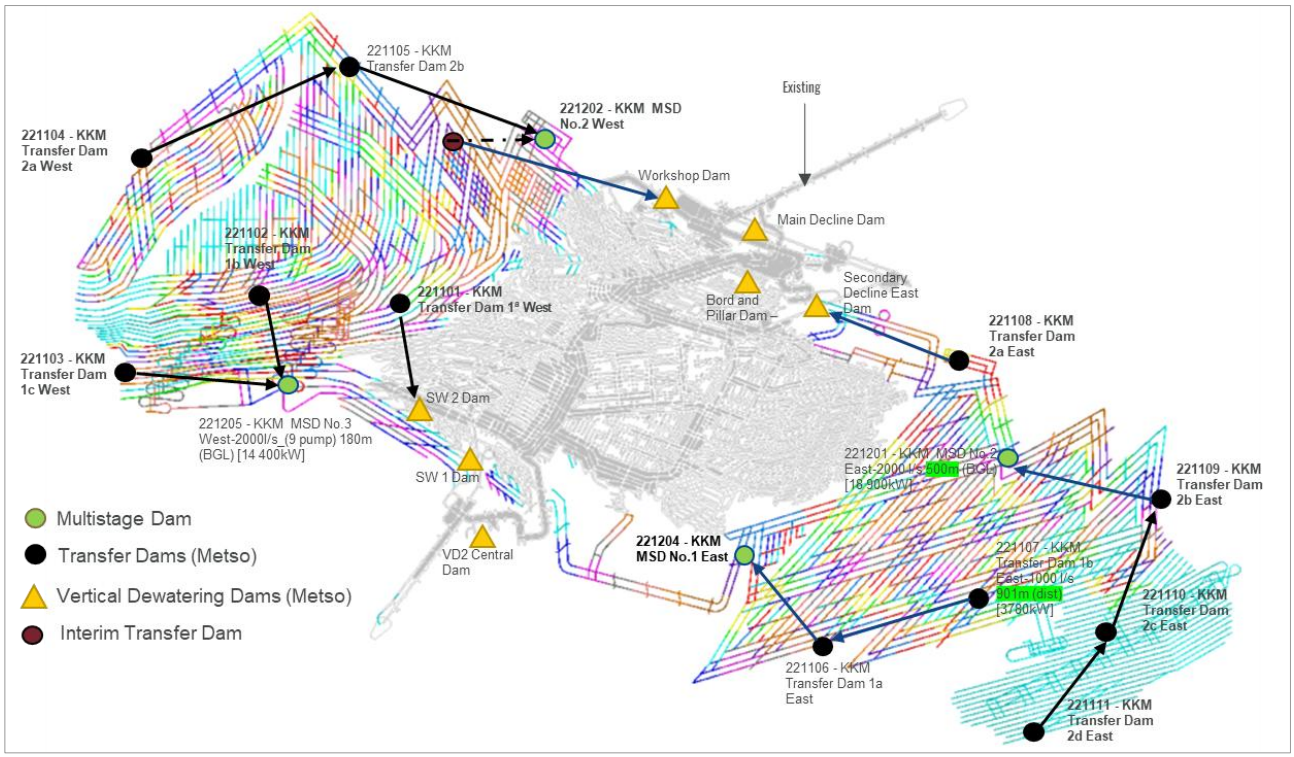
Currently installed dewatering capacity at Kakula mine is 4,200 l/s, and the total is expected to increase to a peak of 11,600 l/s over the LOM. Future development thus requires an additional 7,200 l/s of dewatering capacity. The dewatering configuration for Kakula mine is illustrated in Figure 16.50.

Figure 16.50 Kakula Mine groundwater ingress



Source: WSP 2026

Figure 16.51 Kakula MINE dewatering layout



Source: DRA 2026

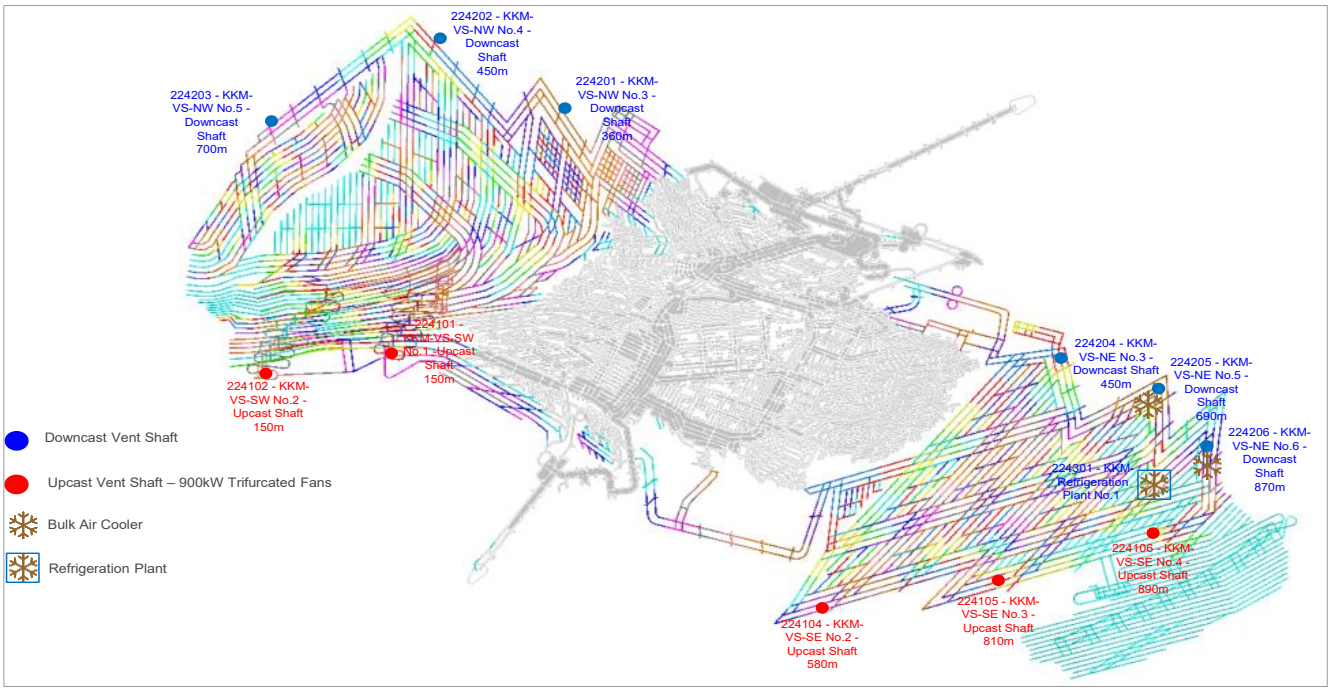
16.5.3 Kakula ventilation strategy

The orebody will be accessed from the surface via the Kakula north and south twin declines, as shown in Figure 16.52. Fresh air will be supplied to the mine from the declines and six 6.1mØ intake ventilation shafts located in the north of the orebody. Guided by ventilation controls, the air will flow through the perimeter drives to ventilate the development and stoping areas

After ventilating the production areas, the air will return to the surface through the KKM-VS-SE No.3, KKM-VS-SE No.4, KKM-VS-SW No.1 and KKM-VS-SW No.2 upcast shafts. A maximum of four upcast fans will be required, but all upcast shafts will be required at different phases throughout the LoM.

Figure 16.52 shows the mine layout with the location of ventilation shafts, with the associated refrigeration locations at the downcast shafts

Figure 16.52 Kakula Mine layout with ventilation shaft locations



Source: DRA Global 11/11/2025

In general, the mine will be supplied with fresh air from intake ventilation shafts located in the north of the mine and via perimeter drives. Ventilation will be extracted through four exhaust shafts in the south of the mine. The model shows the primary ventilation requirements of 2,415 kg/s at the peak production and a maximum refuge bay distance (MRBD) of 817 m.

16.5.4 Kakula production

The production plan for the remainder of Kakula spans 7 years to produce 51.3 million tonnes of ore and 2.0 million tonnes of copper. The production profile is described below in Table 16.35.

Table 16.35 Kakula production profile

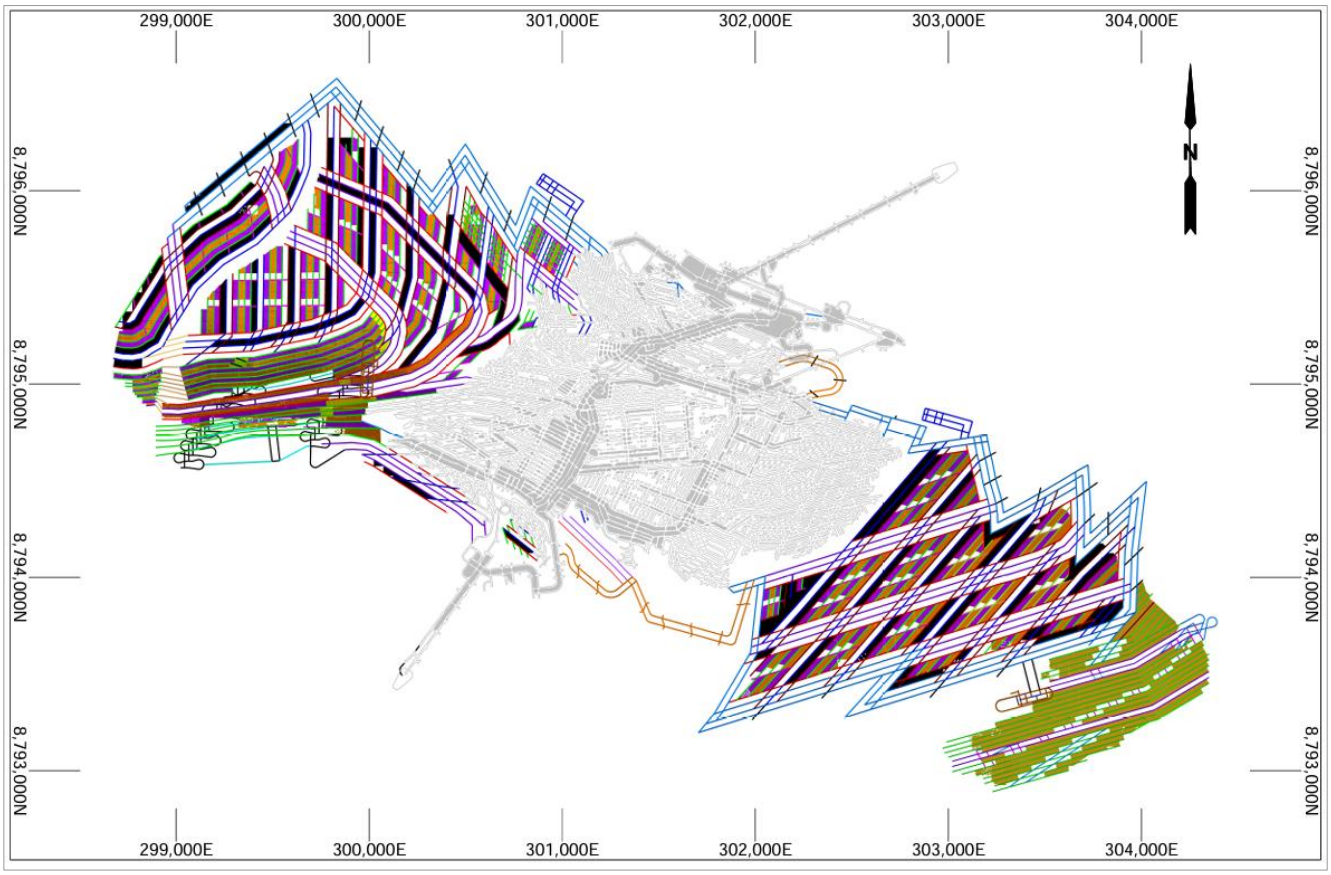
Existing Kamoa Mine	Ore tonnes (Mt)	Grade (%Cu)	Cu metal (Mt)	Peak production (Mtpa)	Steady production (Mtpa)	Production life (Yrs)
Kakula	51.3	3.9%	2.0	8.0	6.5	8

16.5.5 Kakula mine design

The Kakula mining operation primarily uses Drift and Fill methods as descried in Section 16.4.3. Some Cut and Fill mining is planned in the steeper western portions of the deposit.

Two sets of twin access declines, one in the north and one in the south, are designed to access the remaining portions of the Kakula deposit. These declines will provide access, a materials handling trucking route, and infrastructure access such as power and dewatering. (see Figure 16.53).

Figure 16.53 Kakula mine design



Source: KCSA 2026

16.5.6 Recommendations and next steps

Development of the North and South twin declines plus perimeter trunk roads to access undeveloped mining blocks, dewatering sites and to provide haulage routes for production should remain the priority for the eastern portion of Kakula.

Access declines to undeveloped mining blocks and dewatering sites should remain the priority for mining in the western portion of Kakula mine.

16.6 Kamoa 1

16.6.1 Summary of relevant information

Kamoa 1 operations have been established by the development of a portal and underground access into the mine. Production is underway and ore is transported from underground via a conveyor to the Kamoa concentrator. The predominant mining method applied at Kamoa 1 is Drift and Fill mining as described in Section 16.4.3.

16.6.2 Dewatering for Kamoa 1

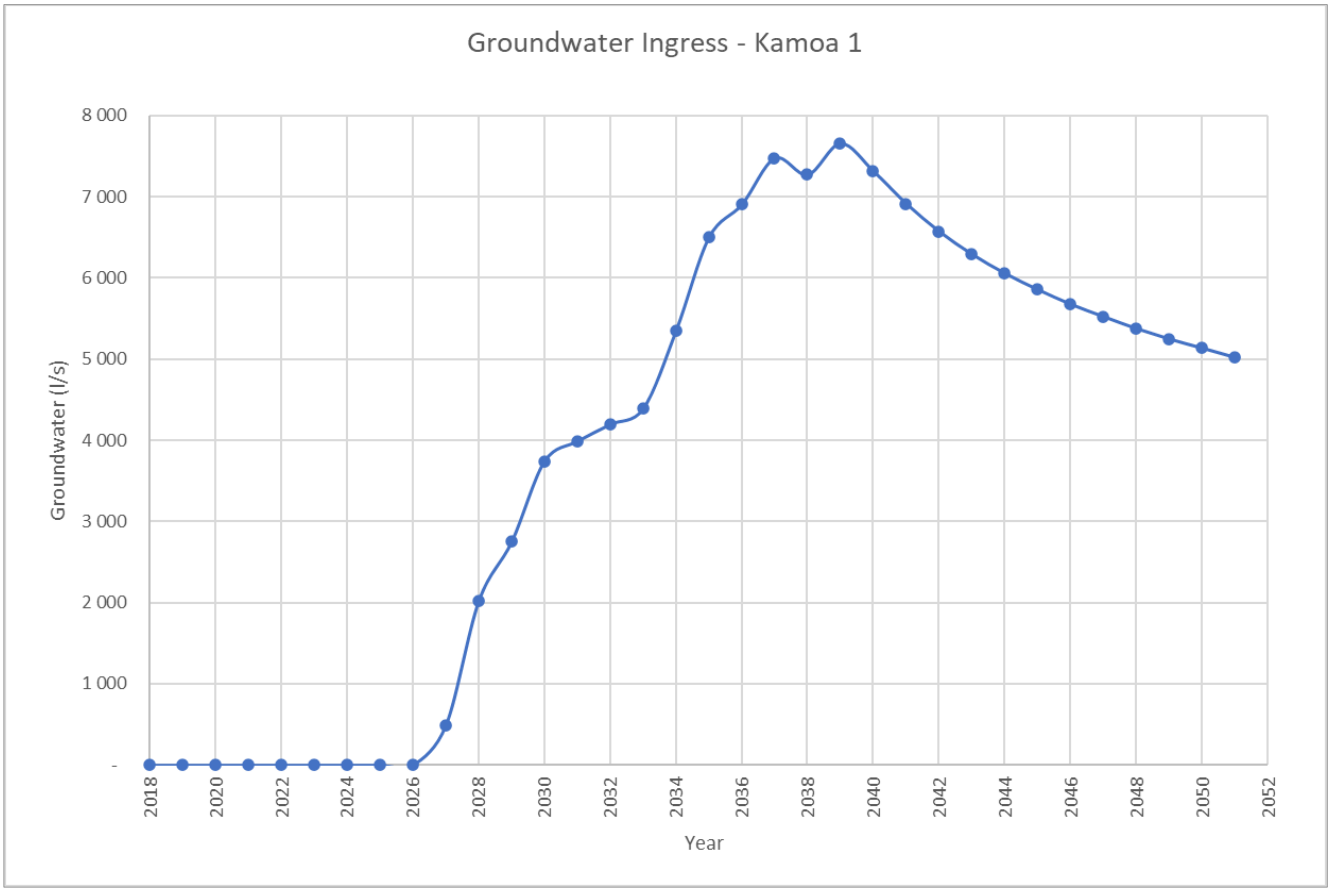
Kamoa 1 will have been equipped with a main decline pumping system which consists of a shaft-bottom transfer dam that pumps water up to an intermediate dam. The intermediate dam is a vertical dewatering

dam designed to discharge water to the surface via a dewatering borehole. The main decline pumping arrangement is capable of handling 1,000 l/s.

In addition to the main decline pumping system, four multistage dewatering pump stations will be established 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 2,000 l/s to surface through dewatering boreholes. Kamoa 1 is expected to experience a peak water inflow rate of 7,700 l/s.

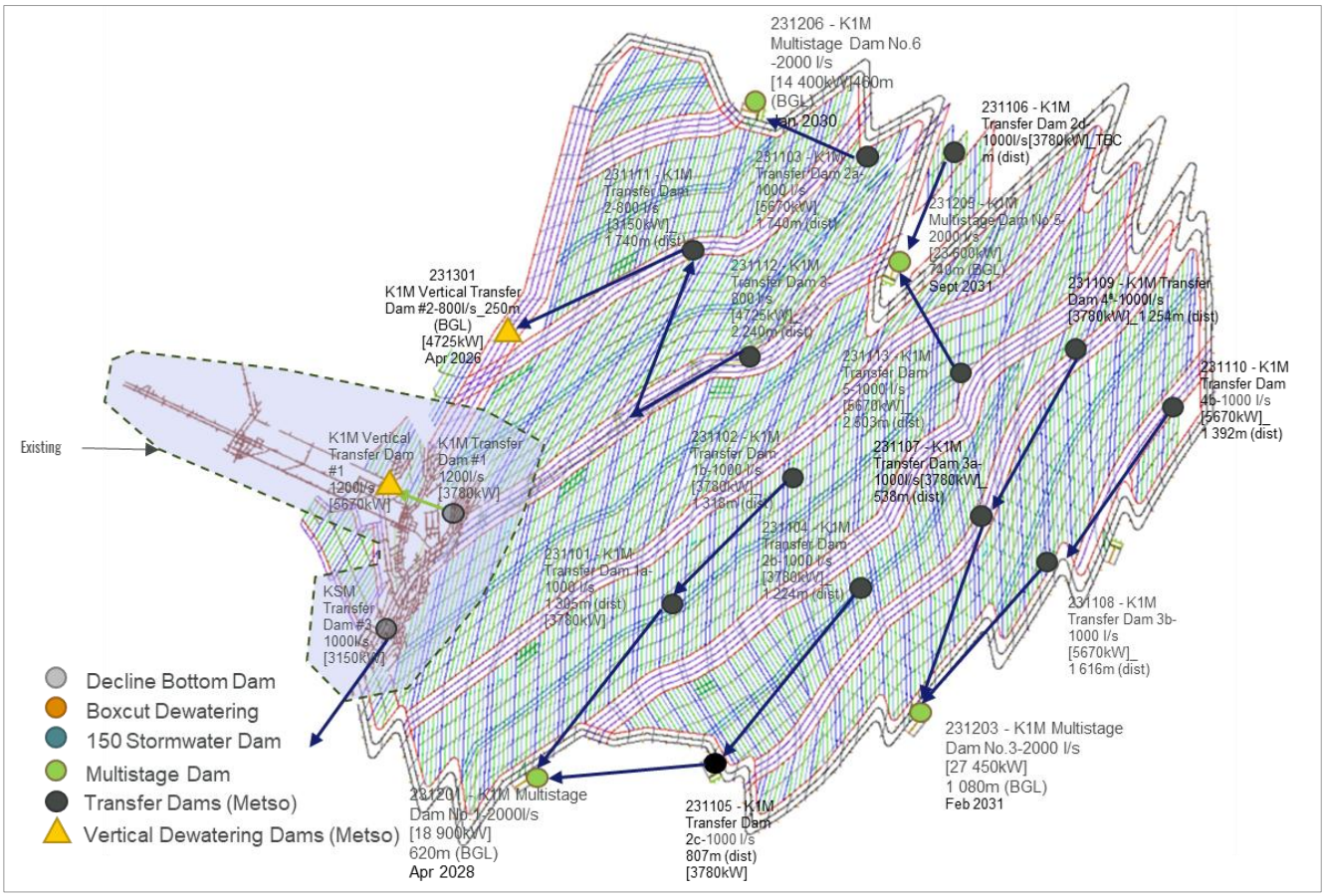
Each multistage dewatering pump station will be fed by two or three transfer dam pump stations. The transfer dam pump stations, indicated as black dots in Figure 16.69, will have the capacity to pump 200 – 1,000 l/s to the main multistage dewatering dams. The transfer dams will be strategically positioned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

Figure 16.54 Kamoa 1 groundwater ingress



Source: WSP 2026

Figure 16.55 Kamoa 1 Mine dewatering layout



Source: WSP 2026

16.6.3 Kamoa 1 ventilation

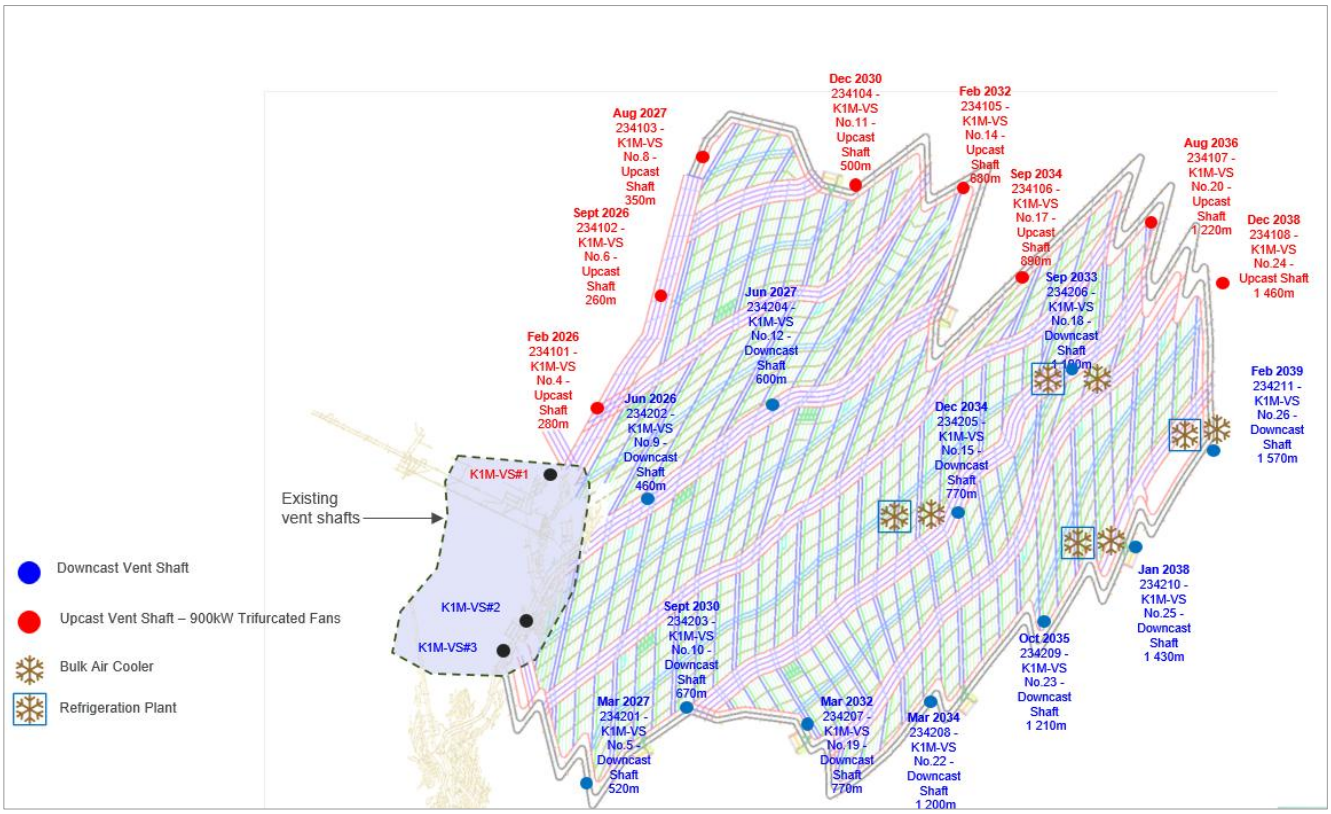
16.6.3.1 Kamoa 1 ventilation strategy

The orebody will be accessed from the surface via the Kamoa 1 twin declines (Access and Conveyor Decline), as shown in Figure 16.56. Fresh air will be supplied to the mine from the declines and eleven 6.1mØ intake ventilation shafts located in the west of the orebody. Guided by ventilation controls, the air will flow through the perimeter drives to ventilate the development and stoping areas.

After ventilating the production areas, the air will return to the surface through the K1M-VS No.14, K1M-VS No.17, K1M-VS No.20 and K1M-VS No.24 upcast shafts. A maximum of four upcast shafts will be required, but all upcast shafts will be required at different phases throughout the LoM.

The conveyor belt will be ventilated as part of the intake but will be equipped so that it can be directly ventilated to return in case of a fire.

Figure 16.56 Kamoa 1 Mine layout with ventilation shaft locations



Source: DRA, 2025

16.6.3.2 Kamoa 1 Primary ventilation requirements

This section describes the primary ventilation circuit (main intake and return) in terms of capacity and distribution. The overall ventilation quantity for this mine is determined by the need to dilute diesel fumes, remove heat (distribute cooling) and dilute blasting fumes during re-entry. It must be noted that the overriding factor here is dilution of diesel fumes.

Primary ventilation requirements were based on diesel exhaust gas dilution at a specific availability rate and utilization factor. The LOM peak airflow requirements are 2,430 kg/s at the production rate of 6.9 Mtpa, and a maximum depth of 1,354 m.

16.6.4 Kamoa 1 production

The production plan for the remainder of Kamoa 1 spans 26 years to produce 103.9 million tonnes of ore and 2.8 million tonnes of copper. The production profile is described below in Table 16.36

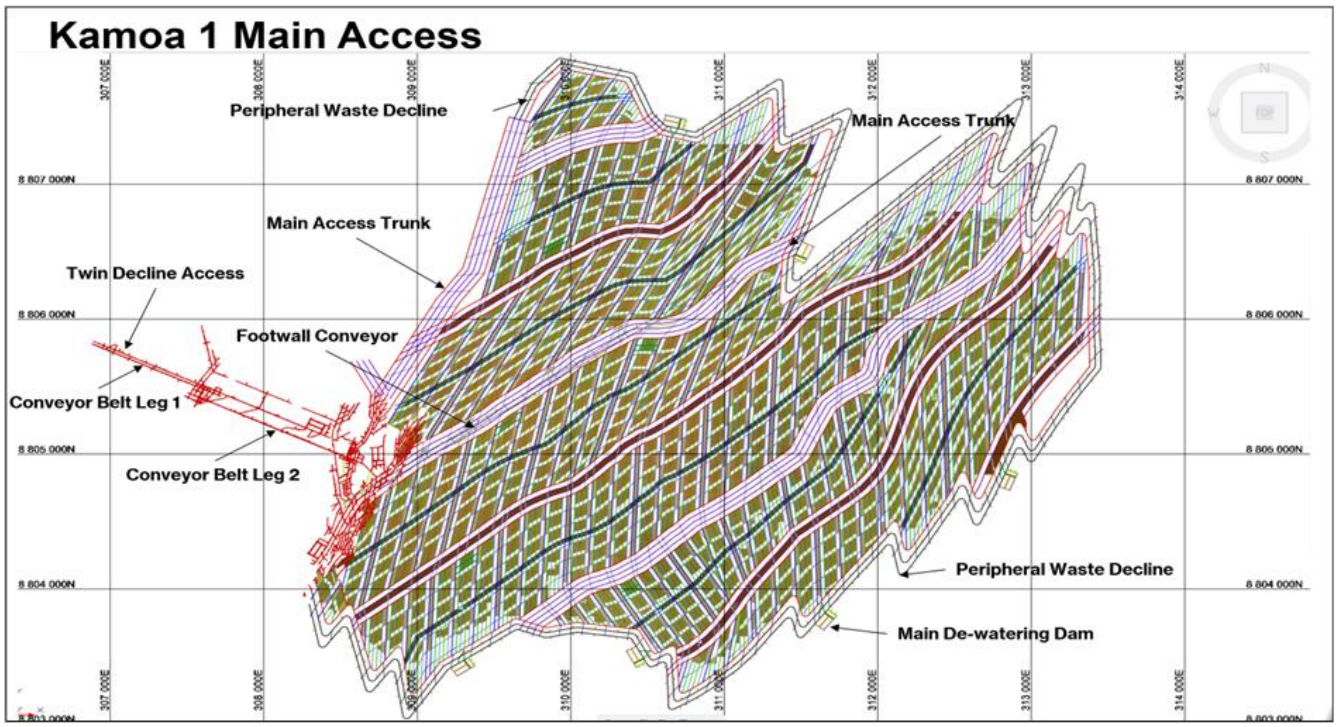
Table 16.36 Kamoa 1 production profile

Existing Kamoa Mine	Ore tonnes (Mt)	Grade (%Cu)	Cu metal (Mt)	Peak production (Mtpa)	Steady production (Mtpa)	Production life (Yrs)
Kamoa 1	103.9	2.7	2.8	5.5	4.0	26

16.6.5 Kamoa 1 Mine Design

Main access into the ore body is created through box cuts with service, trucking and conveying declines. Upon intersecting the ore body main access drives are created and driven on apparent dip, which opens the ore reserves. The on-reef main access development is paired in twins and in some main access areas, triple declines are created on apparent dips of between 7 and 9 degrees and opens the mining areas to access ore. An example of the mine design and lay-out for Kamoa 1 mine can be seen in Figure below. Each cluster of trunk drives through the ore bodies are flanked by ledging drives which allows for stoping access.

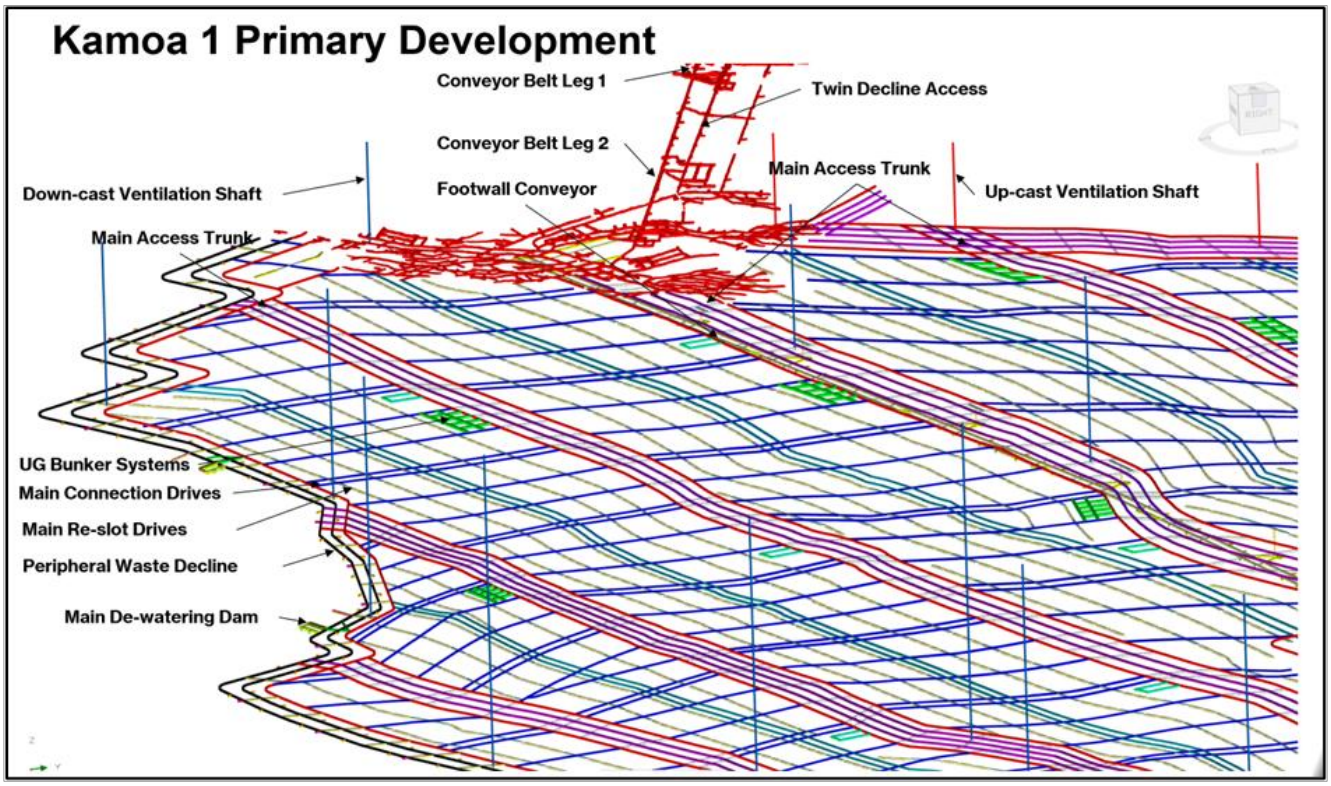
Figure 16.57 Kamoa 1 Main Access Development



Source: KCSA 2026

Main connection drives are driven between main access drives, which are typically spaced between 300 m to 400 m on strike. The main connection drives are also twinned, and connections are typically spaced between 100 m and 150 m apart on dip. The Primary Development is sized at 6 m wide, and the heights vary between 5 m and 6 m depending on the mineralized height of the ore body, see Figure 16.58. Primary Development is prioritized as Priority ends to enable the timely build-up of ore reserves. Primary development delineates the ore blocks and opens these on the best cut to allow the Secondary development process.

Figure 16.58 Primary Development Footprint and Lay-out for Kamoa 1



Source: KCSA 2026

16.6.6 Modifying factors

Initial modifying factors applied to the determining mining shapes through a mine shape optimizer (MSO) process are described below. A minimum development dimension of 5 m (H) x 5 m (W) has been applied to all designs, which incorporates lower grade material through the process when ore thickness is below these dimensions. Mining shapes of D&F excavations have been established to a minimum thickness of 2.5 m, with a minimum aggregate grade of 1.5%TCu.

Table 16.37 Kamoa 1 Mining Dilution and Recovery

Activity	Dilution Factor	Mining Recovery Factor
Horizontal Development	1.05	0.95
Vertical Development	1.05	0.95
Stope	1.1	0.9

In addition to specific mining shape dilution and recovery estimates, an analysis of the resultant mine design resolution was conducted after the designs had been completed for areas where the thickness of the ore was between 2.5 m and 4.0 m. An additional total factor of 10% dilution and 5% mining loss was applied to reflect an allowance for additional drifts and mining floor stripping for remote loader access in longhole benched portions of the ore. Resulting in a combined dilution factor of approximately 21% and or loss of 15%

16.6.7 Recommendations and next steps

Kamoa 1 is an important contributor to production for KCSA. Primary focus should be on development of mining blocks to enable expansion of the mining front in accordance with a stable extraction sequence requirement.

16.7 Kamoa 2

16.7.1 Summary of relevant information

The Kamoa 2 deposit is under development with underground access gained from Kamoa 1 and from independent declines developed from a surface box-cut and portal. The Kamoa 2 deposit has three distinct regions, namely Kamoa 2 West (Kahala), Kamoa2 Central and Kamoa 2 East.

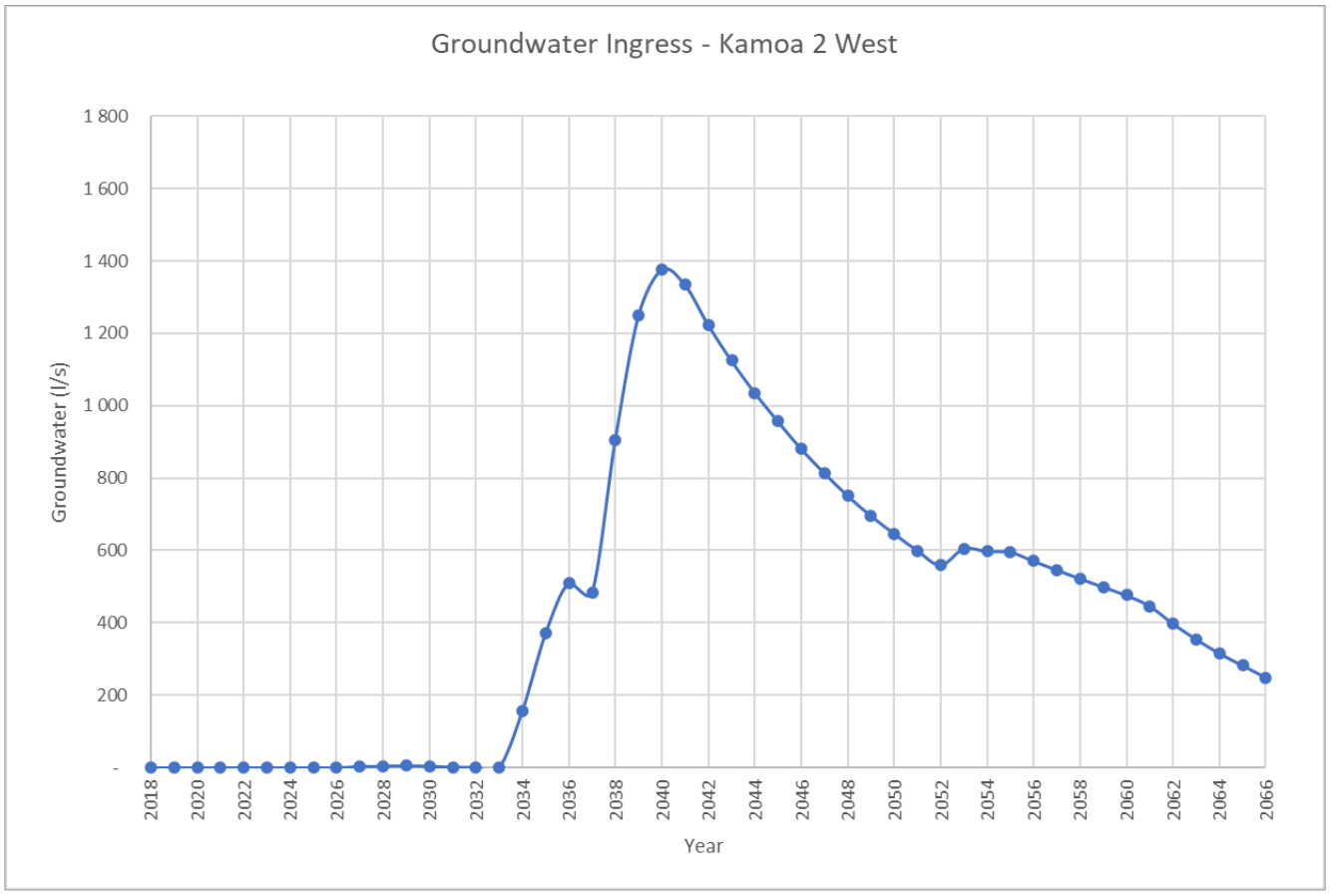
16.7.2 Dewatering for Kamoa 2

Kamoa 2 will have a main decline pumping system at each decline bottom, which consists of a vertical dewatering dam which will pump out to surface via dewatering boreholes. The main decline pumping arrangement can handle 400 l/s each.

In addition to the main decline pumping system, two additional vertical dewatering pump stations will be located along the orebody – indicated by the green dots in Figure 16.69. The western vertical dewatering pump station will be capable of pumping 1,000 l/s to surface and the southern vertical dewatering pump station will be capable of pumping 400 l/s to surface.

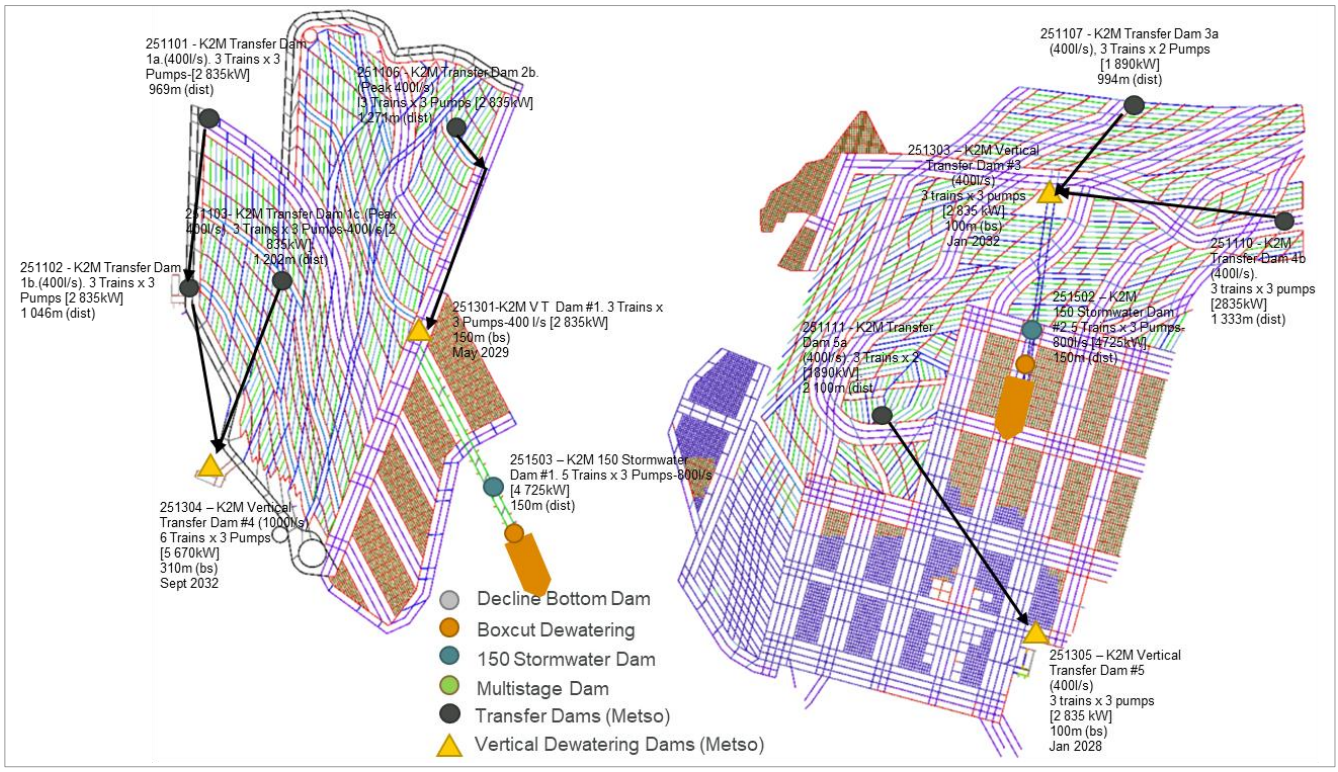
Kamoa 2 will have an expected peak water inflow rate of 1,400 l/s with Kamoa 2 West contributing 1,380 l/s and Kamoa 2 Central and Kamoa 2 East contributing the remainder. The latter two Kamoa 2 mining areas are effectively dewatered by Kamoa 1. However, pumping allowance is made for isolated water bodies not reporting to Kamoa 1.

Figure 16.59 Kamoa 2 West groundwater ingress



Source: WSP 2026

Figure 16.60 Kamoa 2 Mine Dewatering Layout



Source: DRA 2026

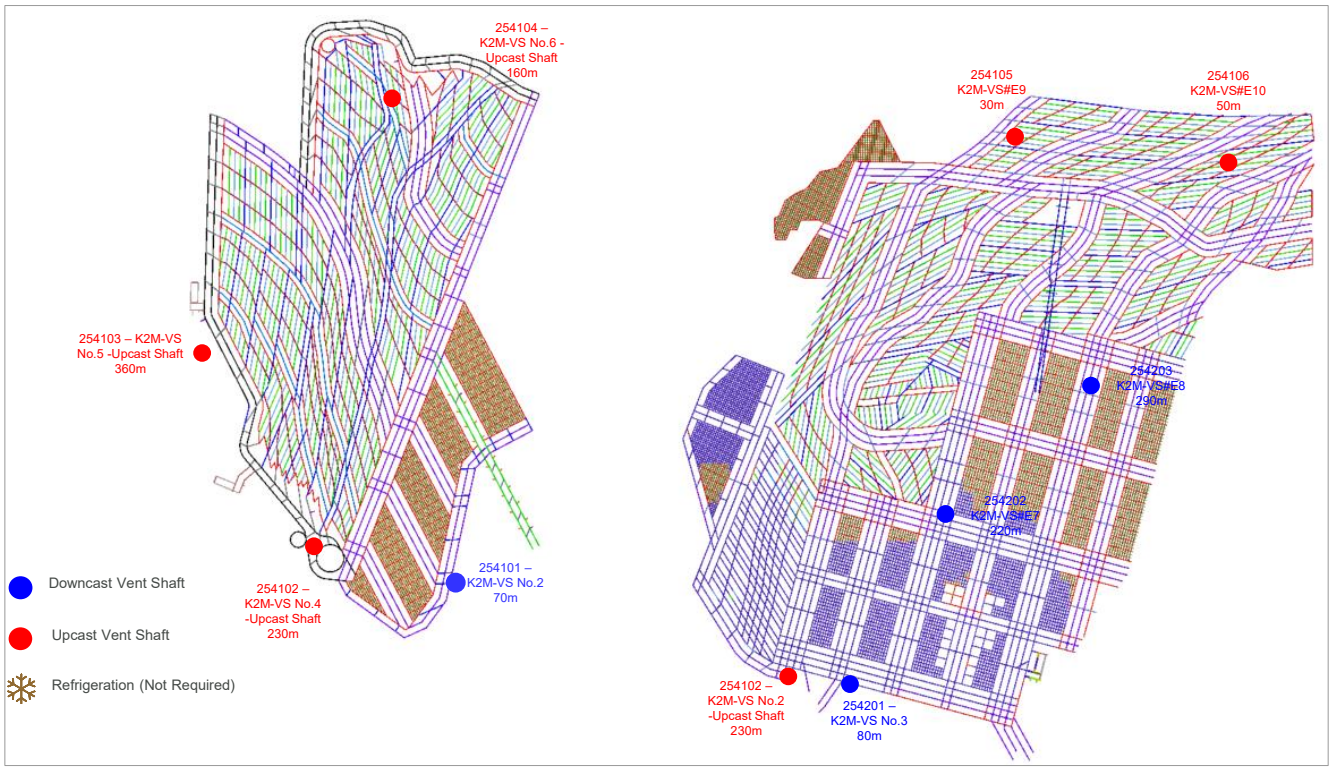
16.7.3 Kamoa 2 ventilation

The orebody will be accessed from the surface via the Kamoa 2 east and west twin declines, as shown in Figure 16.61. Fresh air will be supplied to the mine from the declines and three 6.1mØ intake ventilation shafts located in the east of the orebody. Guided by ventilation controls, the air will flow through the perimeter drives to ventilate the development and stoping areas.

After ventilating the production areas, the air will return to the surface through the K2M-VS No.2, K2M-VS No.4, K2M-VS No.5, K2M-VS No.6, K2M-VS No.9 and K2M-VS No.10 upcast shafts. Only three upcast shafts will be required at any time, but all upcast shafts will be required at different phases throughout the LoM.

Figure 16.61 shows the mine layout with intake and return ventilation shafts.

Figure 16.61 Kamoa 2 Mine layout with ventilation shaft locations



Source: DRA Global 2025

16.7.4 Kamoa 2 Primary ventilation requirements

The overall ventilation quantity for this mine is determined by the need to dilute diesel fumes, remove heat (distribute cooling) and dilute blasting fumes during re-entry. It must be noted that the overriding factor here is diluting diesel fumes.

The model shows the primary ventilation requirements of 1,540 kg/s at the production rate of 5.2 Mtpa, and a depth of 256 mbs.

16.7.5 Kamoa 2 production

The production plan for the remainder of Kamoa 2 spans 27 years to produce 78.1 million tonnes of ore and 2.0 million tonnes of copper. The mine design has been divided into two distinct areas, namely Kamoa 2 Central and East, and Kamoa 2 West. The production profile is described below in Table 16.38

Table 16.38 Kamoa 2 production profile

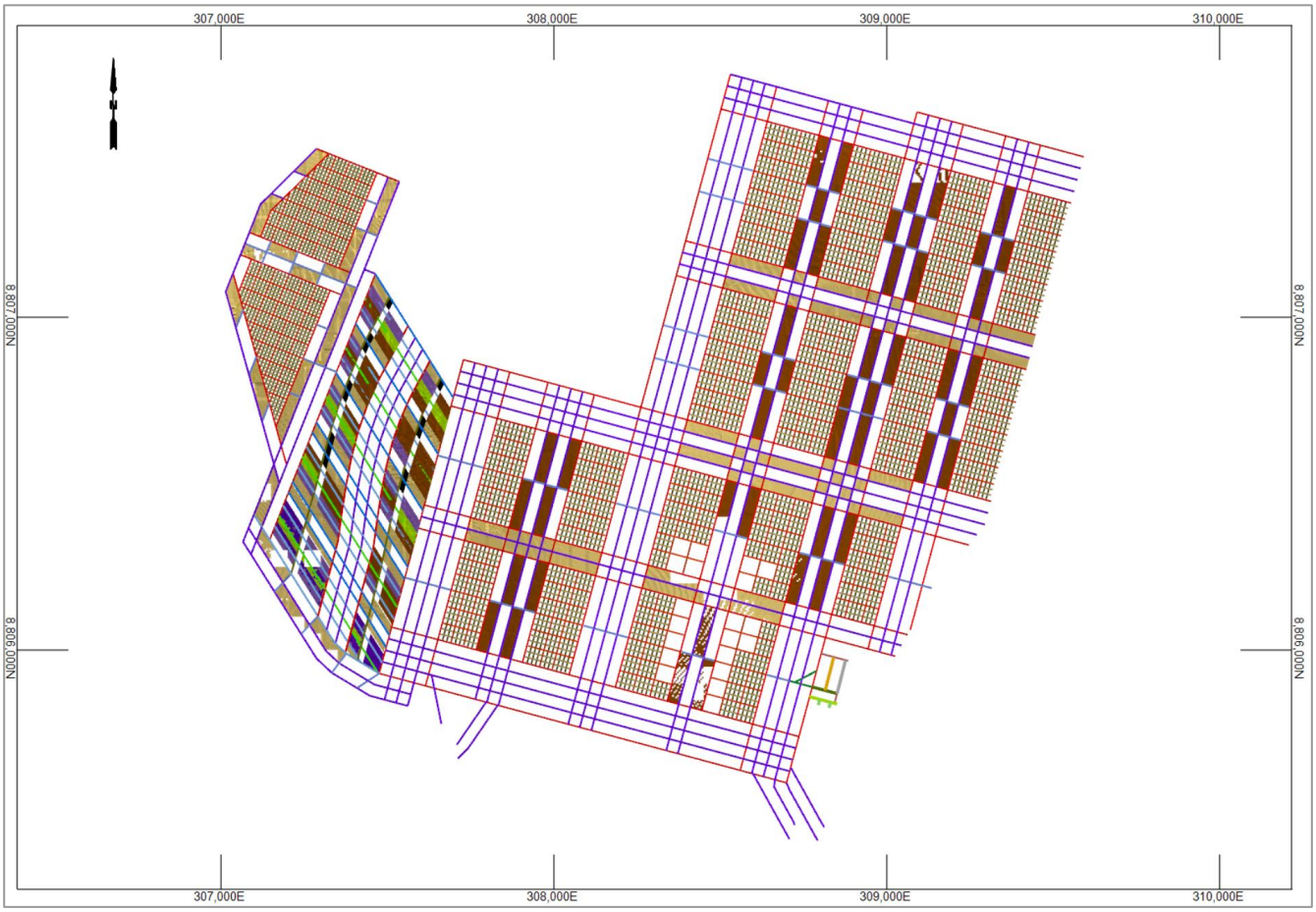
Existing Kamoa Mine	Ore tonnes (Mt)	Grade (%Cu)	Cu metal (Mt)	Peak production (Mtpa)	Steady production (Mtpa)	Production life (Yrs)
Kamoa 2 Central and East	57	2.6	1.5	3.3	2.1	27
Kamoa 2 West	21	2.5	0.5	2.8	1.8	12
Total Kamoa 2	**78**	**2.5**	**2.0%**	**-**	**-**	**-**

16.7.6 Mine design

The three portions of the Kamoa 2 deposit are mined using a combination of Drift and Fill mining in the thicker, sloped area and room and pillar mining in the flat areas. The Kamoa 2 West orebody outcrops on surface and the Kahal Declines are developed on ore, from the box-cut. Near surface mining (within 70 m of Surface) have been excluded from an underground mining plan and will be assessed in the future considering various surface mining options.

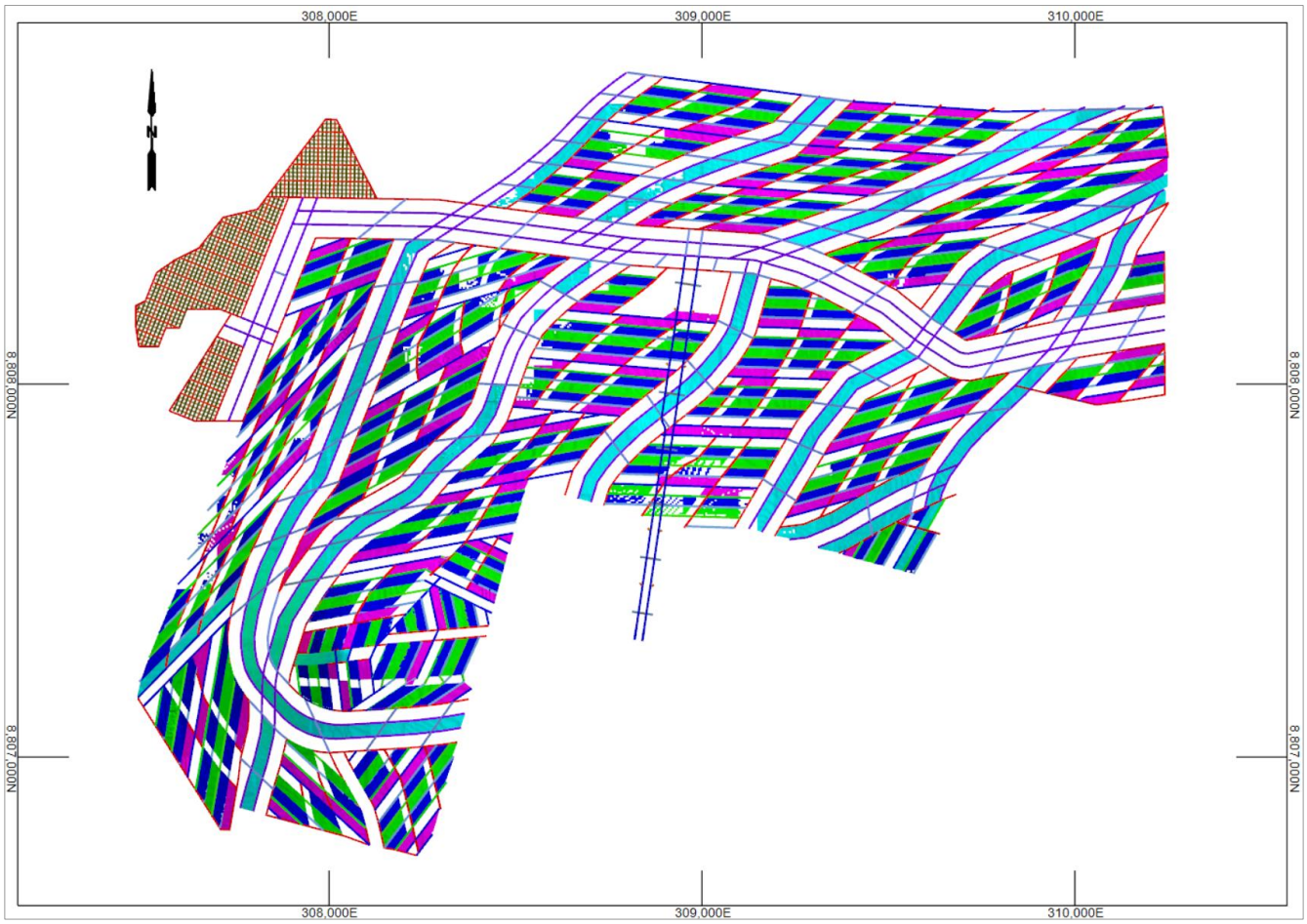
The Mine design for the various sections of Kamoa 2 are illustrated in Figure 16.62, Figure 16.63, Figure 16.64, Figure 16.65, and Figure 16.66

Figure 16.62 Kamoa 2 Central



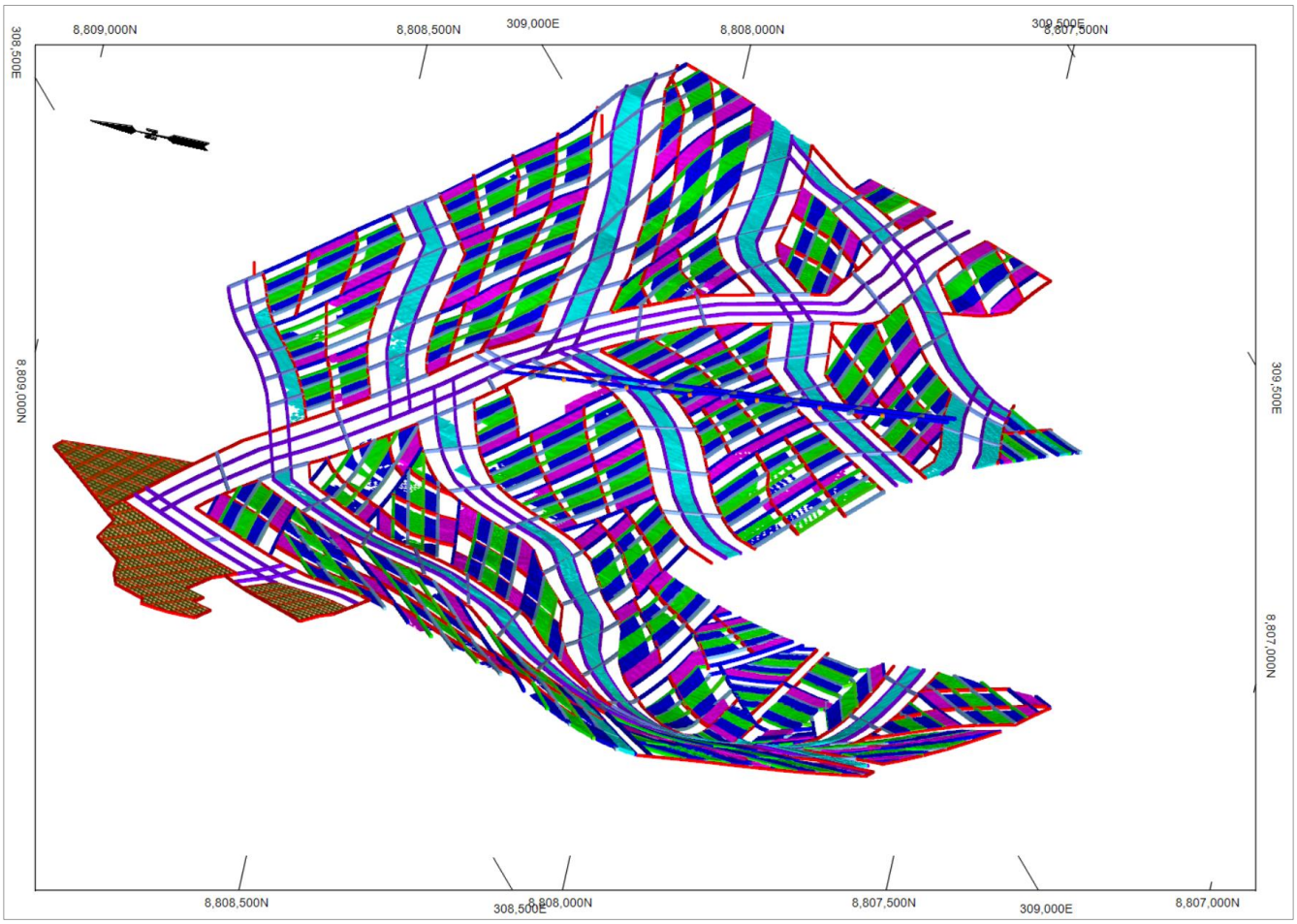
Source: KCSA 2026

Figure 16.63 Kamoa 2 East plan view



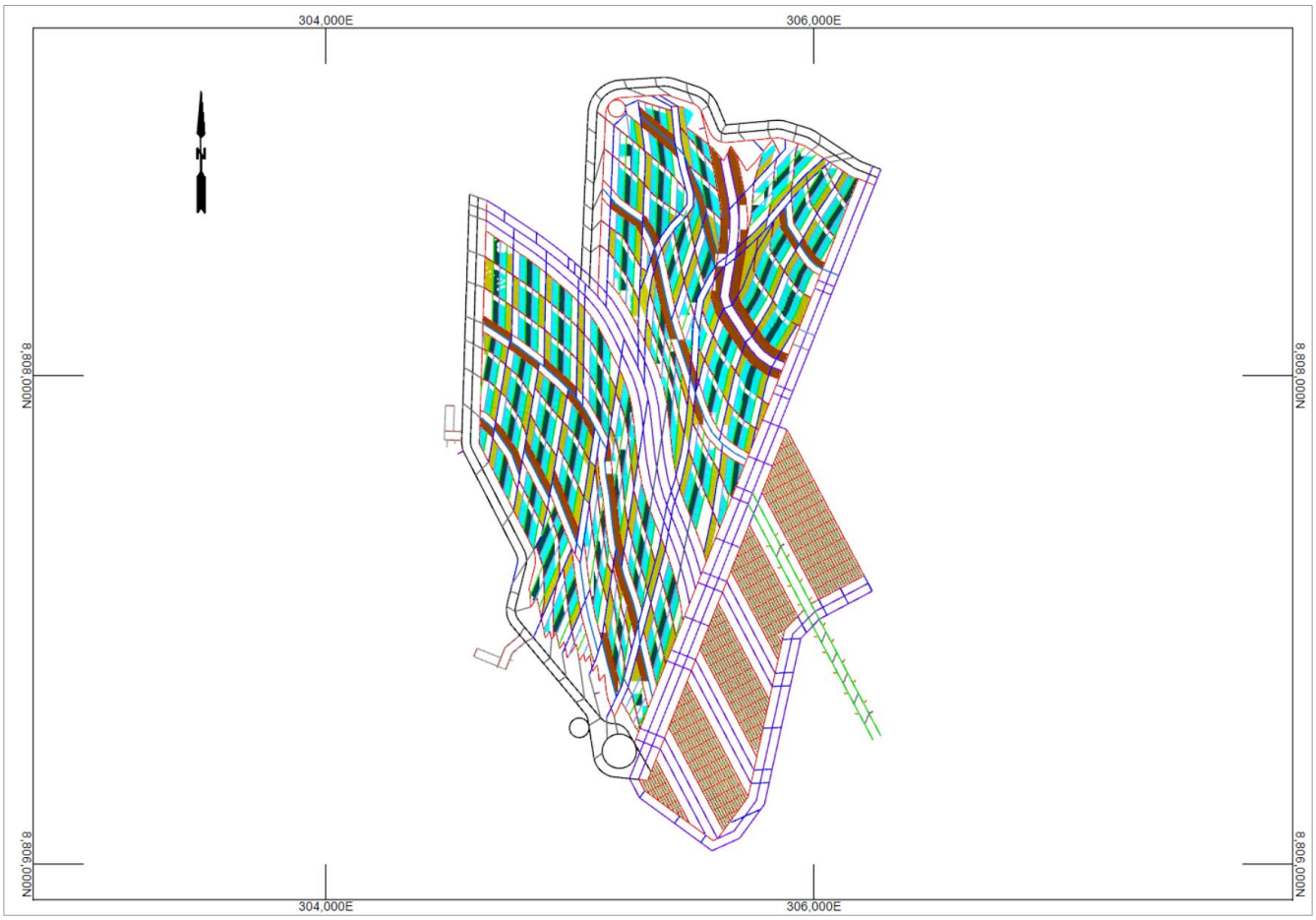
Source: KCSA 2026

Figure 16.64 Kamoa 2 East isometric view



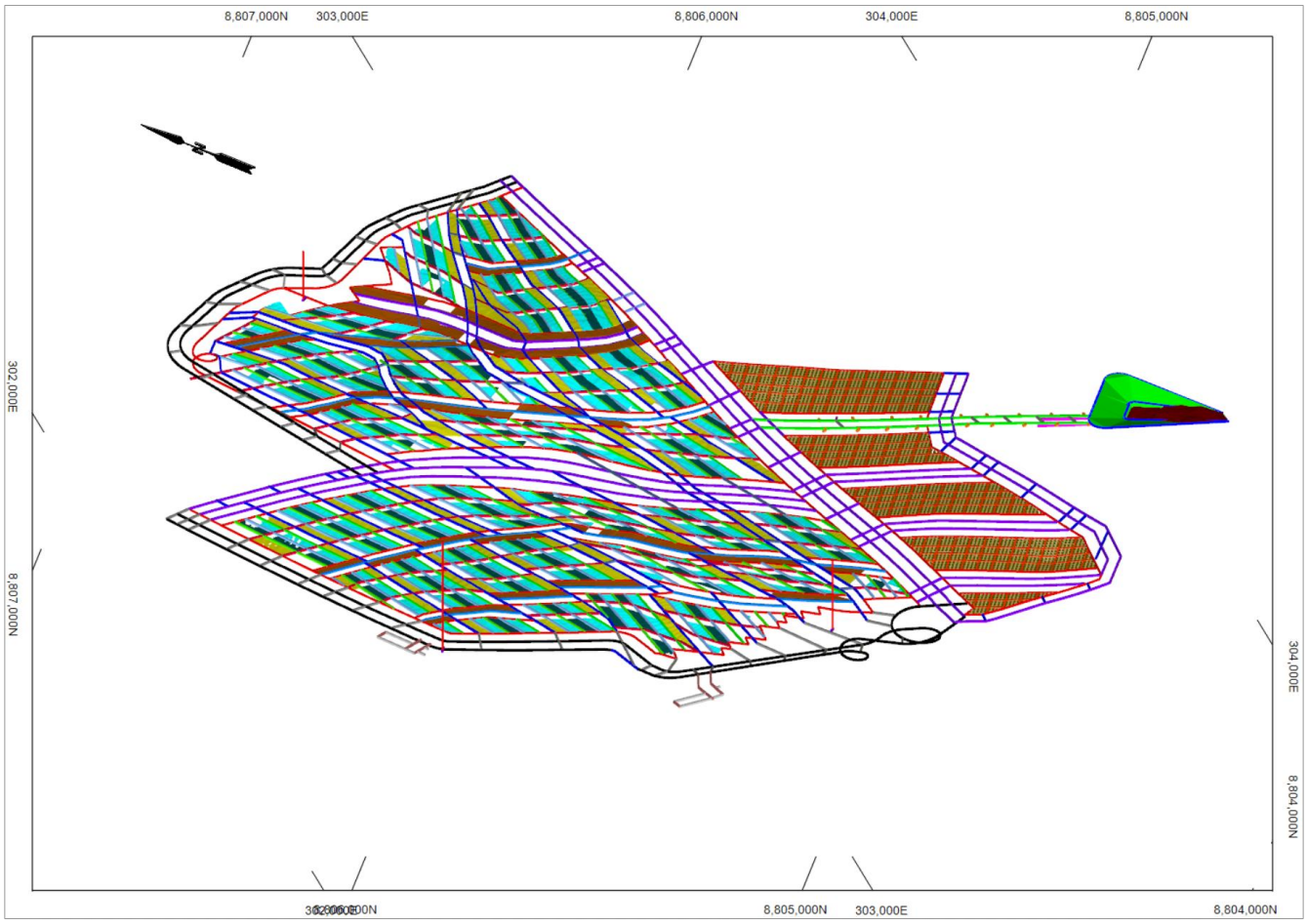
Source: KCSA 2026

Figure 16.65 Kamoa 2 West plan view



Source: KCSA 2026

Figure 16.66 Kamoa 2 West isometric view



Source: KCSA 2026

16.7.6.1 Production summary

The priority for the mining schedule at Kamoa 2 West (Kahala) in 2026 is to establish underground access through the Kahala declines and access high grade to support production.

16.7.7 Modifying factors

Initial modifying factors applied to the determining mining shapes through a mine shape optimizer (MSO) process are described below. A minimum development dimension of 5 m (H) x 5 m (W) has been applied to all designs, which incorporates lower grade material through the process when ore thickness is below these dimensions. Mining shapes of D&F excavations have been established to a minimum thickness of 2.5 m, with a minimum aggregate grade of 1.5%TCu. Mining dilution and recovery applied to Kamoa 2 are listed in Table 16.39

Table 16.39 Kamoa 2 Mining Dilution and Recovery

Activity	Dilution Factor	Mining Recovery Factor
Horizontal Development	1.05	0.95
Vertical Development	1.05	0.95
Stope	1.1	0.9

In addition to specific mining shape dilution and recovery estimates, an analysis of the resultant mine design resolution was conducted after the designs had been completed for areas where the thickness of the ore was between 2.5 m and 4.0 m. An additional total factor of 10% dilution and 5% mining recovery loss was applied to reflect an allowance for additional drifts and mining floor stripping for remote loader access in longhole benched portions of the ore. Resulting in a combined dilution factor of approximately 21% and or loss of 15%.

16.7.8 Recommendations and next steps for Kamoa 2

Development of the Kahala West declines should remain a priority to activities in Kamoa 2.

An assessment of potential open-pit mining of near surface portions of the deposit should be completed. This potential open pit assessment should be integrated into a combined extraction strategy for Kamoa 2.

16.8 Kansoko Sud

16.8.1 Summary of relevant information

Kansoko Sud has been developed and generated production from underground operations for approximately 3 years. The majority of ore production to date has been generated from ore development, with a smaller quantity from drift and fill mining. (See section 16.4.3 for a description of mining methods)

16.8.1.1 Dewatering for Kansoko Sud

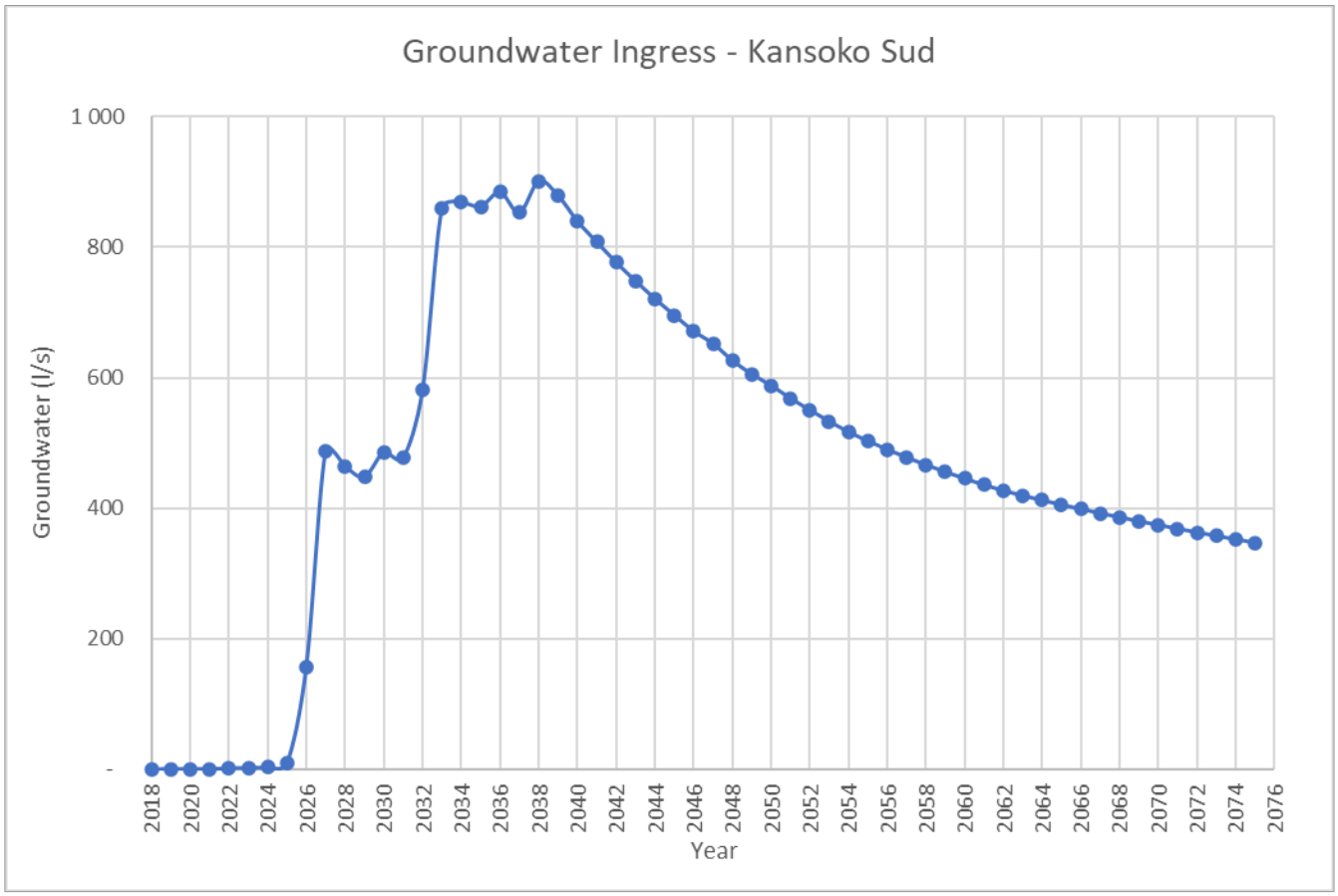
Kansoko Mine is equipped with a main decline pumping system consisting of a shaft-bottom transfer dam that pumps water to surface. The main decline pumping arrangement can handle 1,000 l/s and currently caters for dewatering of all access areas within the existing mine, including water transfer from Kamoa 1 access.

In addition to the main decline pumping system, a vertical transfer pump station has been established near the southern extremity of the orebody. This station allows Sud Mine to transfer water to the same facility and can pump 1,200 l/s to surface through a dewatering borehole. Kansoko Mine (at 200 l/s) and Sud Mine (at 900 l/s) are expected to experience a combined peak water inflow rate of 1,100 l/s.

Sud Mine will require an additional multistage dewatering pump station, which will be established along the southwestern extremity of the orebody. This pump station will be capable of pumping 2,000 l/s to surface through a dewatering borehole.

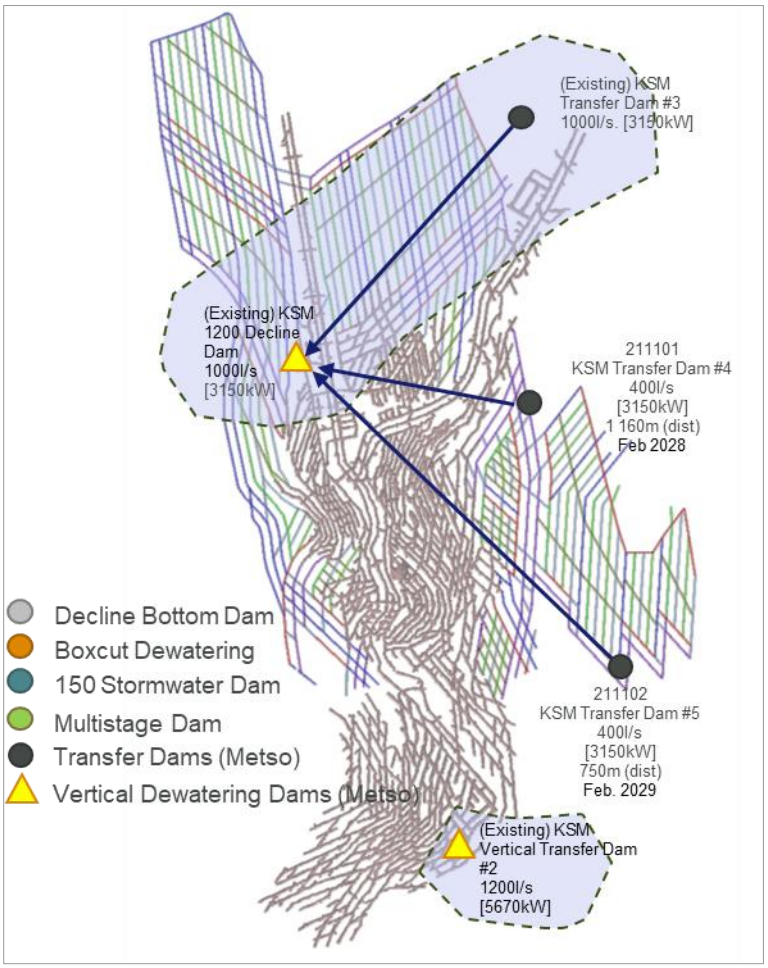
Each vertical transfer pump station and multistage dewatering pump station will be fed by two or three transfer dam pump stations. The transfer dam pump stations will be strategically positioned to cover the required footprint of the mined orebody.

Figure 16.67 Kansoko Sud Groundwater Ingress



Source: WSP2026

Figure 16.68 Kansoko Sud Mine Dewatering Layout



Source: WSP2026

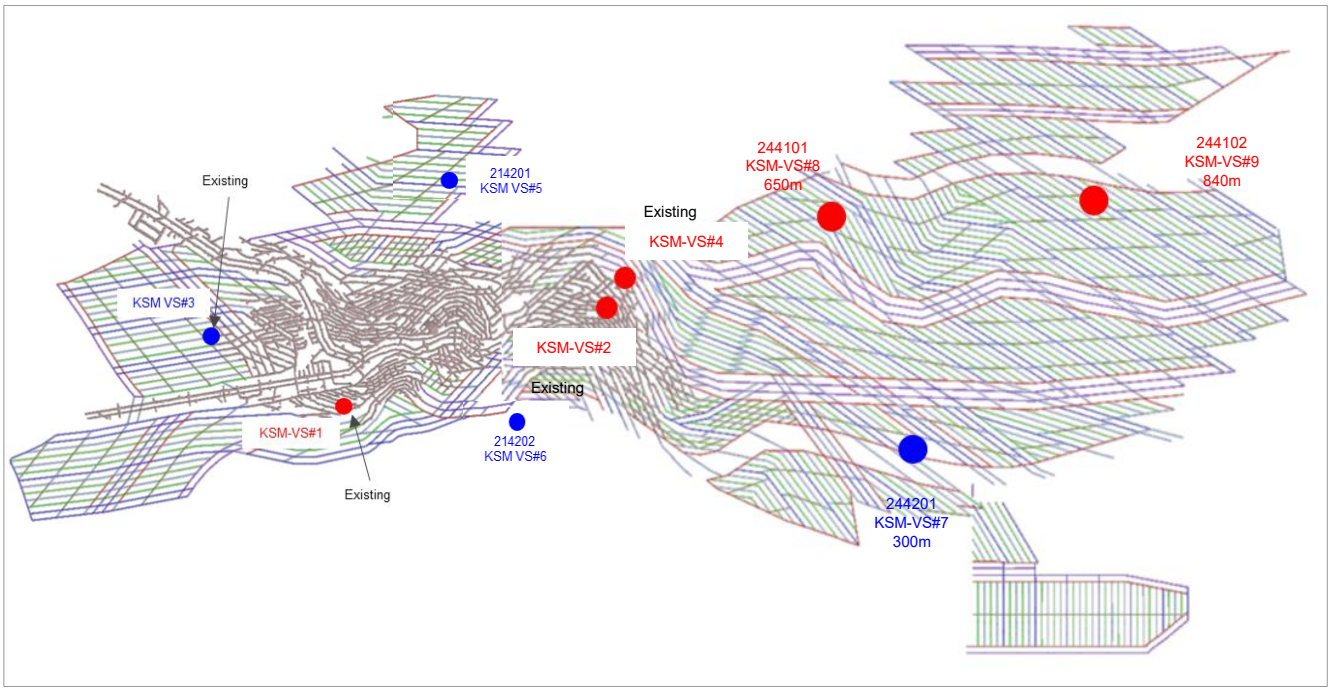
16.8.2 Kansoko Sud ventilation

16.8.2.1 Kansoko Sud ventilation strategy

The orebody will be accessed from the surface via the Kansoko existing twin decline, twin decline number 2 and Single decline number 1, as shown in Figure 16.69. Fresh air will be supplied to the mine from the declines and five 6.1mØ intake ventilation shafts located in the east of the orebody. Guided by ventilation controls, the air will flow through the perimeter drives to ventilate the development and stoping areas.

After ventilating the production areas, the air will return to the surface through the KSM-VS No.2, KSM-VS No.4, KSM-VS No.8, and KSM-VS No.9 upcast shafts. Only two upcast shafts will be required at any time, but all upcast shafts will be required at different phases throughout the LoM.

Figure 16.69 Kansoko Sud Mine layout with ventilation shaft locations



Source: DRA, 2026

16.8.2.2 Kansoko Sud Primary ventilation requirements

The overall ventilation quantity for this mine is determined by the need to dilute diesel fumes, remove heat (distribute cooling) and dilute blasting fumes during re-entry. It must be noted that the overriding factor here is to dilute diesel fumes. The model shows the primary ventilation requirements of 1,211 kg/s at the production rate of 1.2 Mtpa, and a depth of 716 mbs.

16.8.3 Kansoko Sud production

The production plan for the remainder of Kansoko Sud spans 26 years to produce 32.5 million tonnes of ore and 0.9 million tonnes of copper. The production profile is described below in Table 16.40

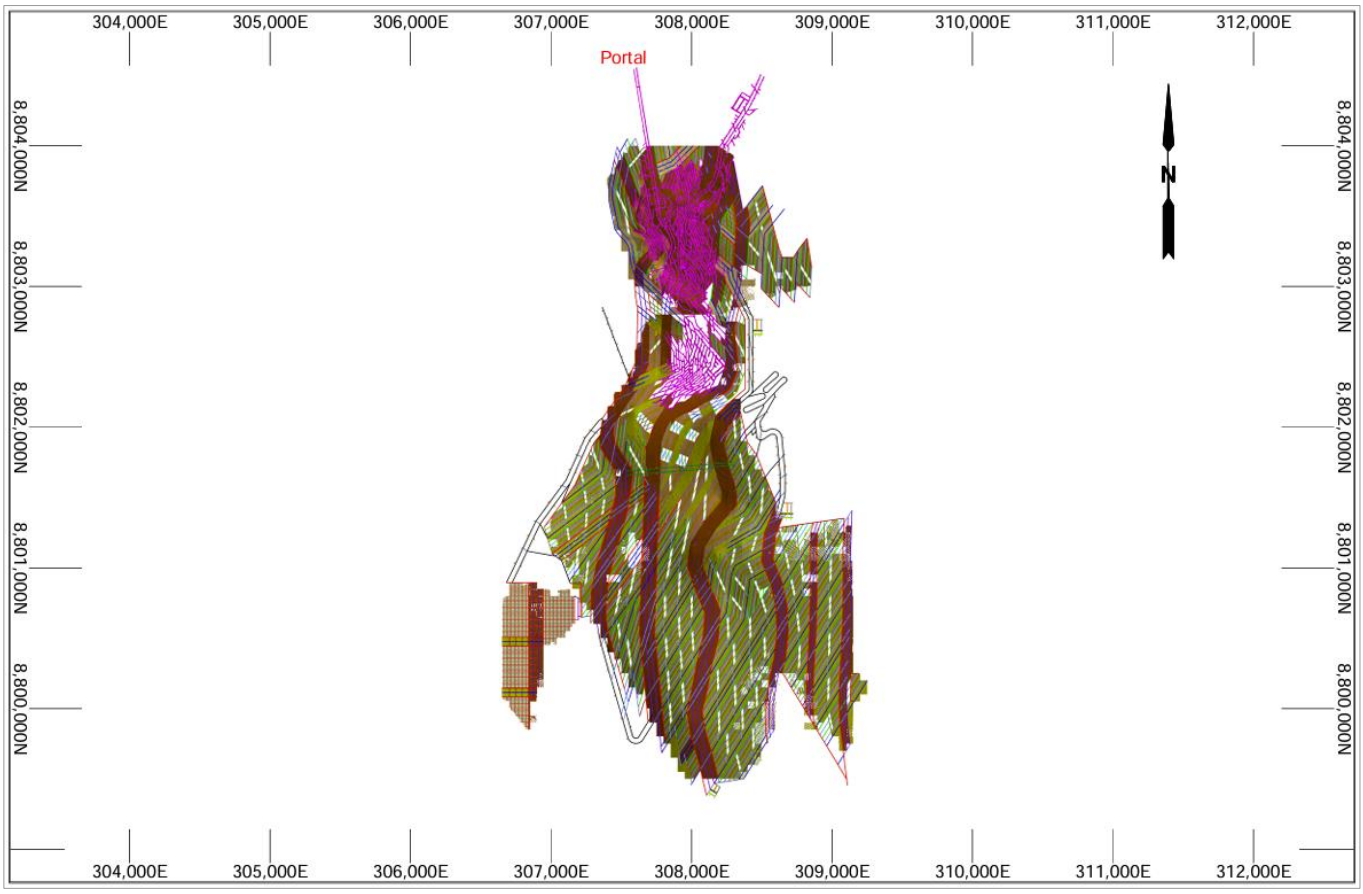
Table 16.40 Kansoko Sud production profile

Existing Kamoa Mine	Ore tonnes (Mt)	Grade (%Cu)	Cu metal (Mt)	Peak production (Mtpa)	Steady production (Mtpa)	Production life (Yrs)
Kansoko Sud	32.5	2.8	0.9	2.6	1.2	26

16.8.4 Mine design

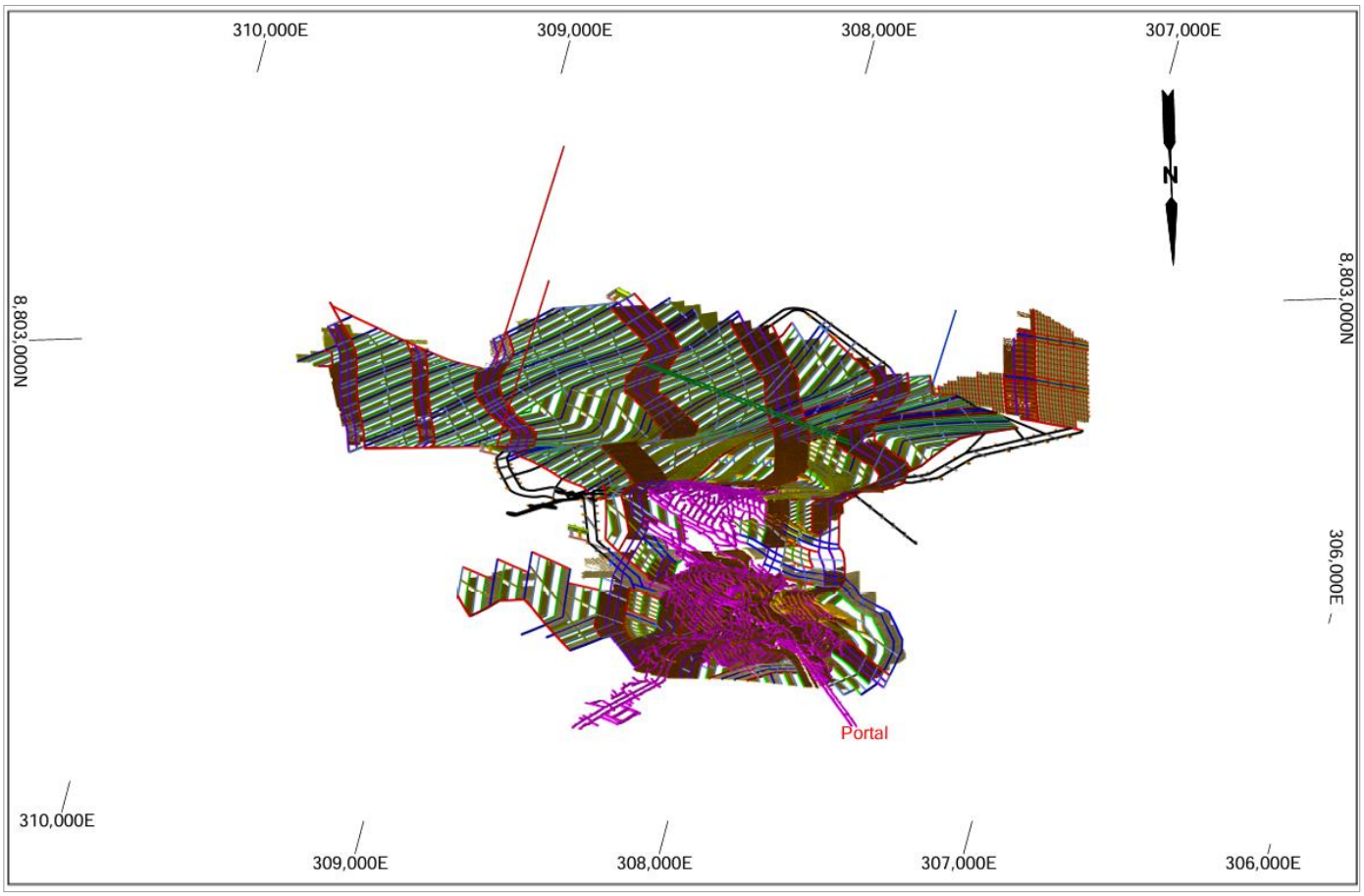
The Kansoko Sud deposit is mined using a combination of Drift and Fill mining in the thicker, sloped area and room and pillar mining in the flat areas. The Mine design for the various sections of Kansoko Sud are illustrated in Figure 16.70 and Figure 16.71.

Figure 16.70 Kansoko Sud plan view



Source: KCSA 2026

Figure 16.71 Kansoko Sud isometric view



Source: KCSA 2026

16.8.5 Modifying factors

Initial modifying factors applied to the determining mining shapes through a mine shape optimizer (MSO) process are described below. A minimum development dimension of 5 m (H) x 5 m (W) has been applied to all designs, which incorporates lower grade material through the process when ore thickness is below these dimensions. Mining shapes of D&F excavations have been established to a minimum thickness of 2.5 m, with a minimum aggregate grade of 1.5%TCu. The mine dilution and recovery parameters applied to Kansoko Sud are described in Table 16.42.

Table 16.41 Modifying factors applied to Kansoko Sud

Activity	Dilution factor	Mining recovery factor
Horizontal development	1.0	1.0
Vertical development	1.0	1.0
Stope	1.1	0.9

In addition to specific mining shape dilution and recovery estimates, an analysis of the resultant mine design resolution was conducted after the designs had been completed for areas where the thickness of the ore was between 2.5 m and 4.0 m. An additional total factor of 10% dilution and 5% mining loss was applied to reflect an allowance for additional drifts and mining floor stripping for remote loader access in

longhole benched portions of the ore. This results in a combined dilution factor of approximately 21% and or loss of 15%.

16.8.6 Recommendations and next steps for Kansoko Sud

The development priority for Kansosko Sud should remain the development of the Sud access decline. This decline will significantly improve the materials haulage route to surface, reducing cycle times, truck wear and overall productivity.

16.9 Kamoa 3

16.9.1 Summary of relevant information

The Kamoa 3 deposit is shallow dipping around 20-25° on average with extremes from 10° to 40°, with some slight undulations in the ore body. The thickness of the considered deposit is between 4.5 m and 12.5 m. This shallow dipping ore body constrains the mining method options to either a drift and fill, cut-and-fill or sidewall slashing method. These methods are generally development intensive, low production rate and high cost.

The chosen method is a bottom-up hybrid of cut and fill mining as described in Section 16.4.

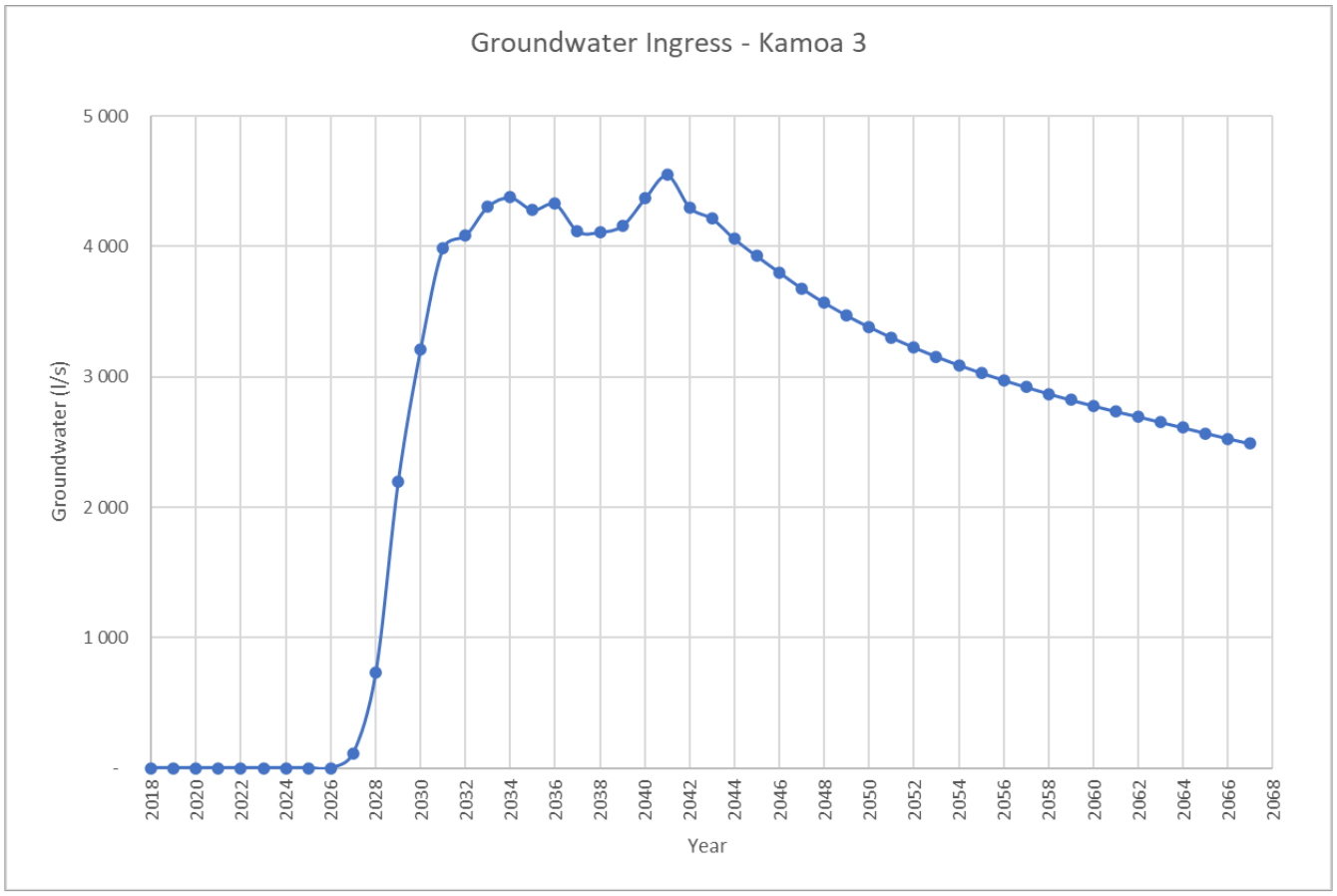
16.9.2 Dewatering for Kamoa 3

Kamoa 3 will have an expected peak water inflow rate of 4,550 l/s with 50% of this volume reached within 5 years of start of mine. Simultaneous spiral development to depth on four mining fronts causes rapid increase of water ingress.

Kamoa 3 will have a main decline pumping system at decline bottom, which consists of two vertical dewatering dams which will pump out to surface via dewatering boreholes. The main decline pumping arrangement is capable of handling 2,400 l/s.

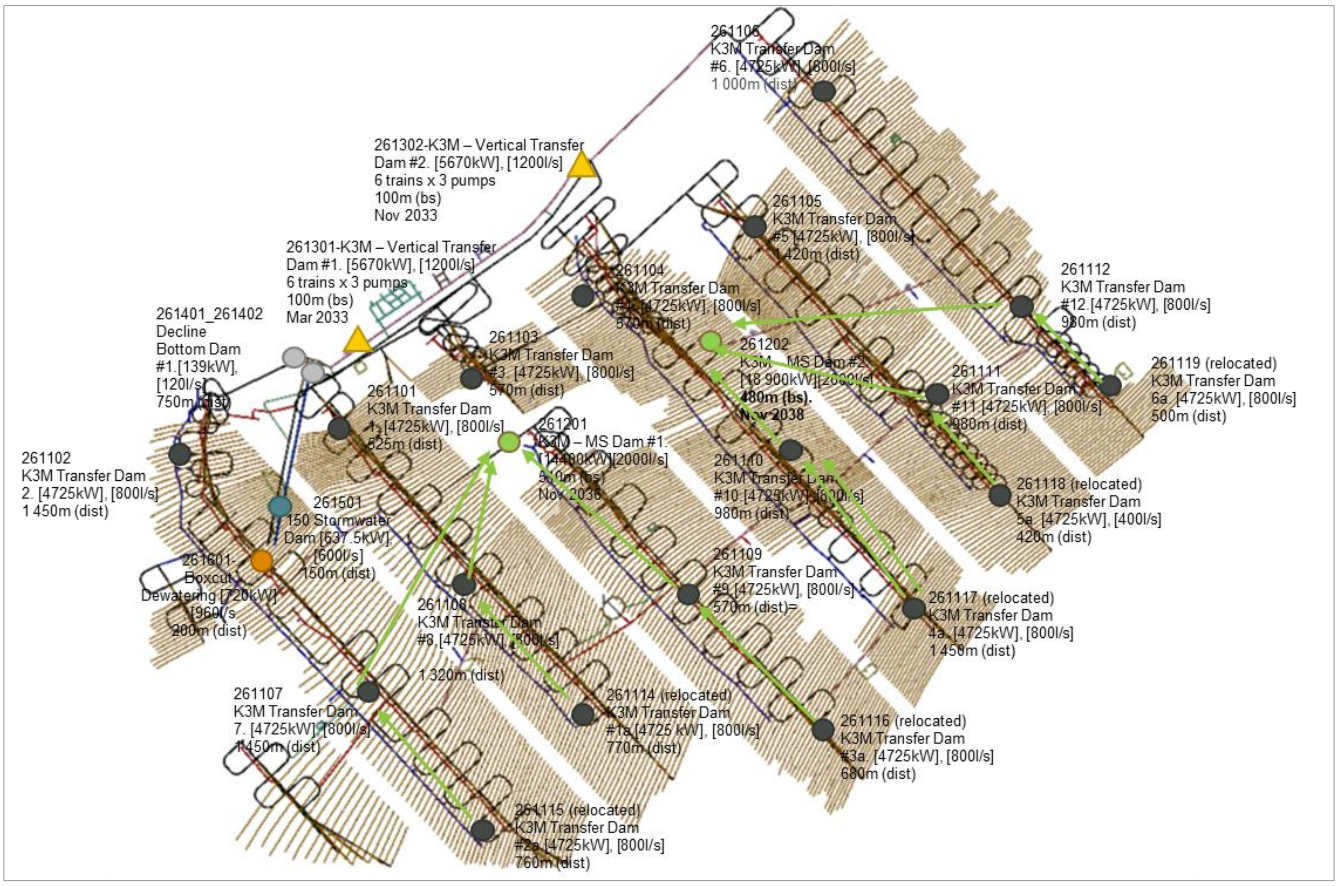
In addition to the main decline pumping system, two additional multistage dewatering pump stations will be located along the Kamoa 3 orebody – indicated by the green dots in Figure 16.73. Each multistage dewatering pump station will be capable of pumping 2,000 l/s to surface through dewatering boreholes.

Figure 16.72 Kamoa 3 groundwater ingress



Source: WSP 2026

Figure 16.73 Kamoa 3 Mine Dewatering Layout



Source: DRA 2026

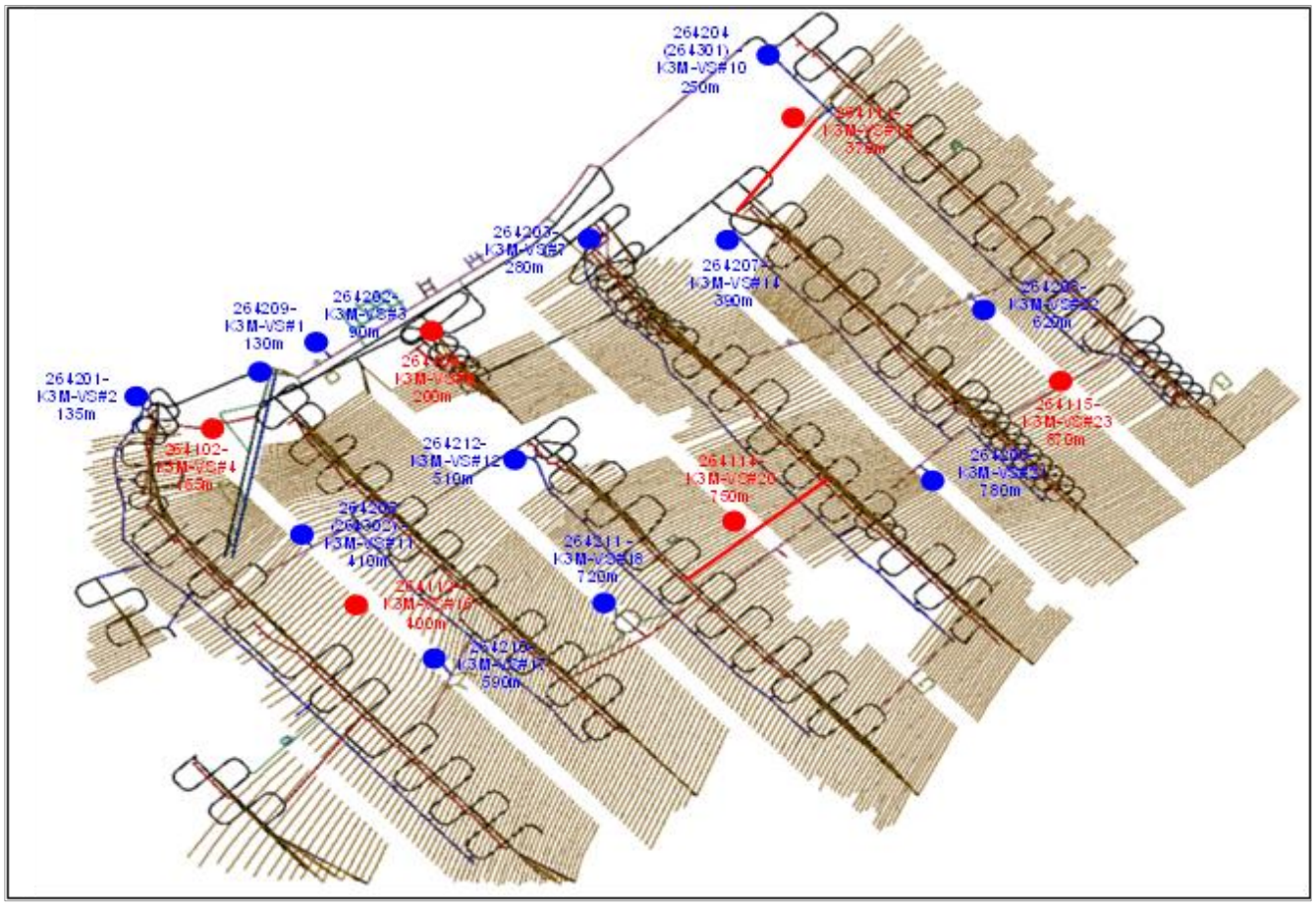
16.9.3 Kamoa 3 ventilation

16.9.3.1 Kamoa 3 Ventilation strategy

The orebody will be accessed from the surface via the Kamoa 3 twin decline, as shown in Figure 16.74 Fresh air will be supplied to the mine from the twin decline and twelve 6.0mØ intake ventilation shafts located across the orebody. Guided by ventilation controls, the air will flow through the perimeter drives to ventilate the development and stoping areas.

After ventilating the production areas, the air will return to the surface through the upcast shafts listed.

Figure 16.74 Kamoa 3 Mine layout with ventilation shaft locations



Source: DRA 2026.

The overall ventilation quantity for this mine is determined by the need to dilute diesel fumes, remove heat (distribute cooling) and dilute blasting fumes during re-entry. It must be noted that the overriding factor here is to dilute diesel fumes. The model shows the primary ventilation requirements of 3,708 kg/s at the peak production rate of 3.0 Mtpa, and a maximum depth of 960 m.

16.9.4 Kamoa 3 production

The Kamoa 3 operation is expected to take 8 years to reach full capacity of around 3.0 Mtpa, it will remain at this level for approximately 7 years and then steadily decline. The subsequent tail period of the mine life for the final 10 years mainly involves the recovery of sill pillars. The Kamoa production profile is shown in Table 16.42.

Table 16.42 Kamoa 3 production profile

Future operations	Ore tonnes (Mt)	Grade (%Cu)	Cu metal (Mt)	Peak production (Mtpa)	Steady production (Mtpa)	Life of production (Yrs)
Kamoa 3	57.6	2.4	1.4	3.5	2.4	21

16.9.4.1 Production summary

Production stopes are designed with a minimum height of 4.5 m and a maximum height of 7.5 m. The stope shapes are cut between contours of 7.5 m or 5 m, depending on the dip of the ore, and ensure that no stopes are wider than 15 m. Flatter regions generally have wider stopes.

Each stoping block, as shown by purple in Figure 16.78 is an independent working area. The sequence of extraction within each block is from the bottom-up. As each independent stoping block is accessed, it begins a new sequence. The final retreat sequence involves extraction of the sill pillars (red) once all material above them is mined.

The Kamoa 3 equipment and task rates validated with the AMC SmartData™ benchmarking database is shown in Table 16.43.

Table 16.43 Schedule equipment and task rates

Equipment or task	Rate
Jumbo	140 m/month
Production Drill	170 m/day
Raisebore	2 m/day
Stoping	800 t/day
Cable Bolter	150 m/day
Ore Drive or Cut-and-Fill Access	2 m/day
Lateral Development (Other)	90 m/month

16.9.4.2 Modifying factors

The modifying factors applied to Kamoa 3 design activities are shown in Table . For the stope recovery factors, they are dependent on the thickness of the ore. The minable shapes generated originally did not follow the 7.5 m geotechnical stope height restriction. Recovery has therefore been reduced proportionally for each increment above 7.5 m ore thickness. Dilution in these thicker zones is also reduced as a greater portion of the bogging horizon can be mined in ore.

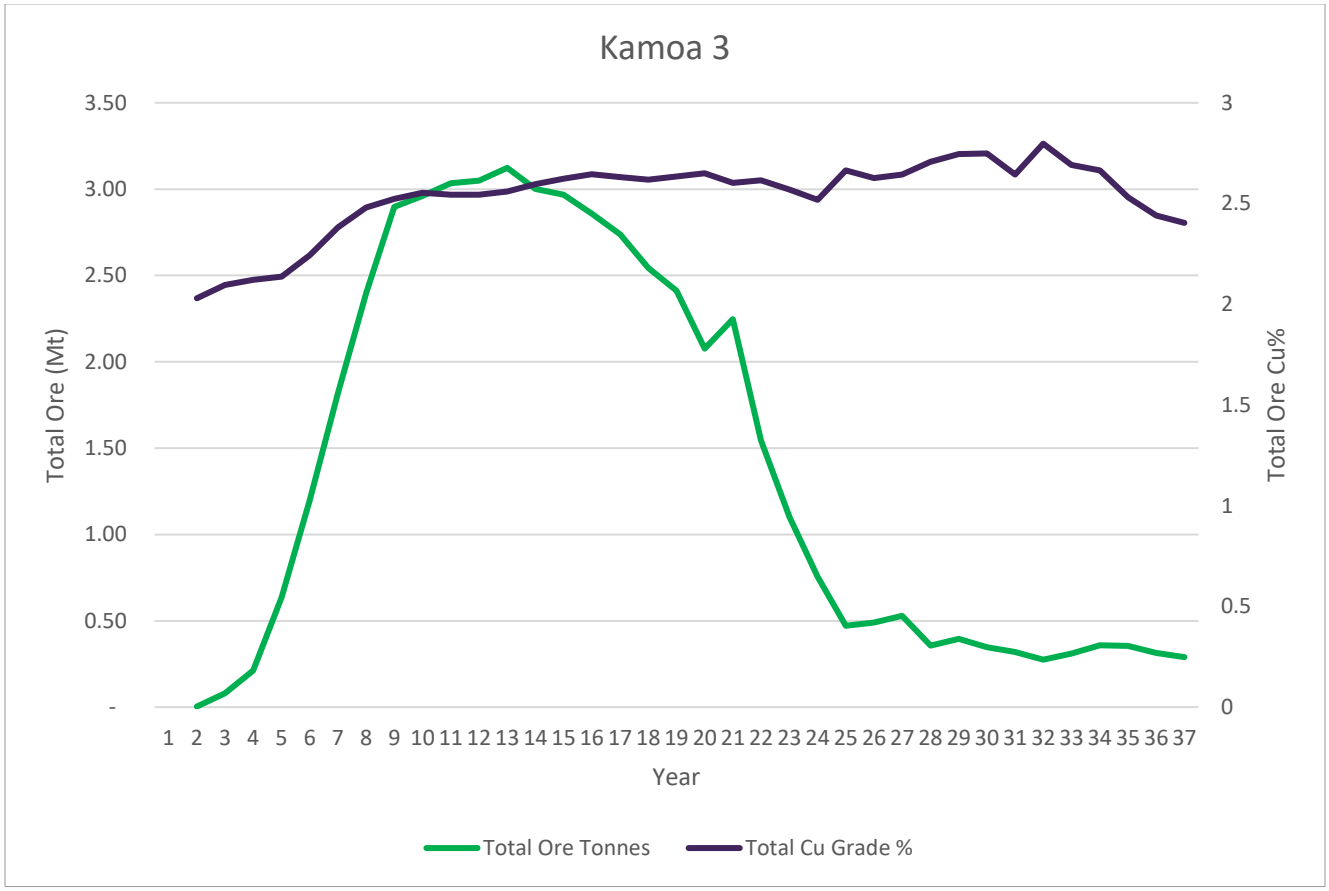
Table 16.44 Kamoa 3 Modifying factors

Activity	Dilution factor	Mining recovery factor
Horizontal development	1.05	0.95
Vertical development	1.0	1.0
Stope < = 7.5 m high	1.2	0.9
Stope thickness [7.5 m, 8.5 m]	1.1	7.5 / thickness x 0.9
Stope thickness > 8.5 m high	1.0	7.5 / thickness x 0.9

16.9.4.3 Production profile

The Kamoa 3 production profile and resulting copper grades are shown in Figure 16.75.

Figure 16.75 Kamoa 3 ore production profile

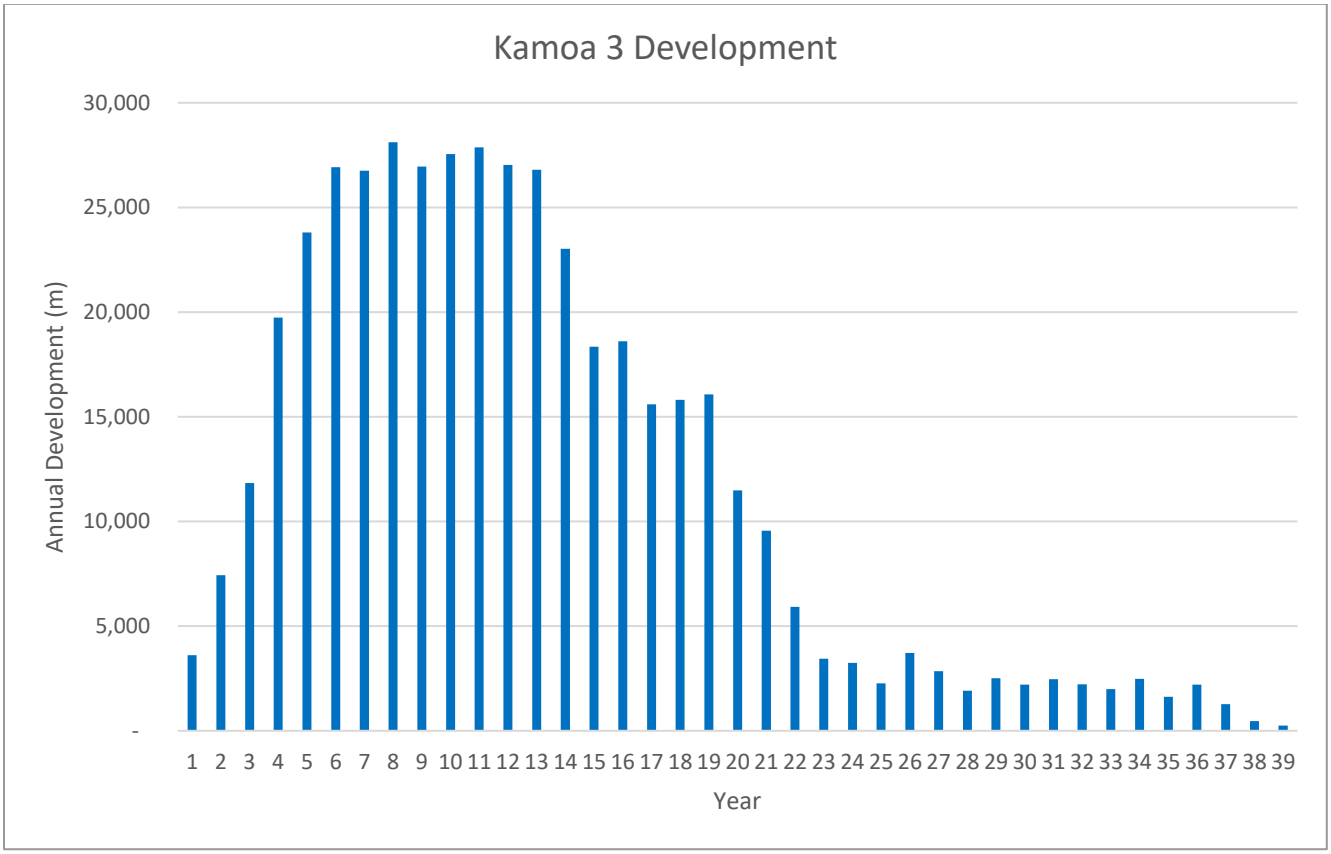


Source: AMC 2026

16.9.5 Kamoa 3 development and infrastructure

The Kamoa 3 development advance profile is shown in Figure 16.76.

Figure 16.76 Kamoa 3 annual development advance profile



Source: AMC2026

16.9.5.1 Basis of Design

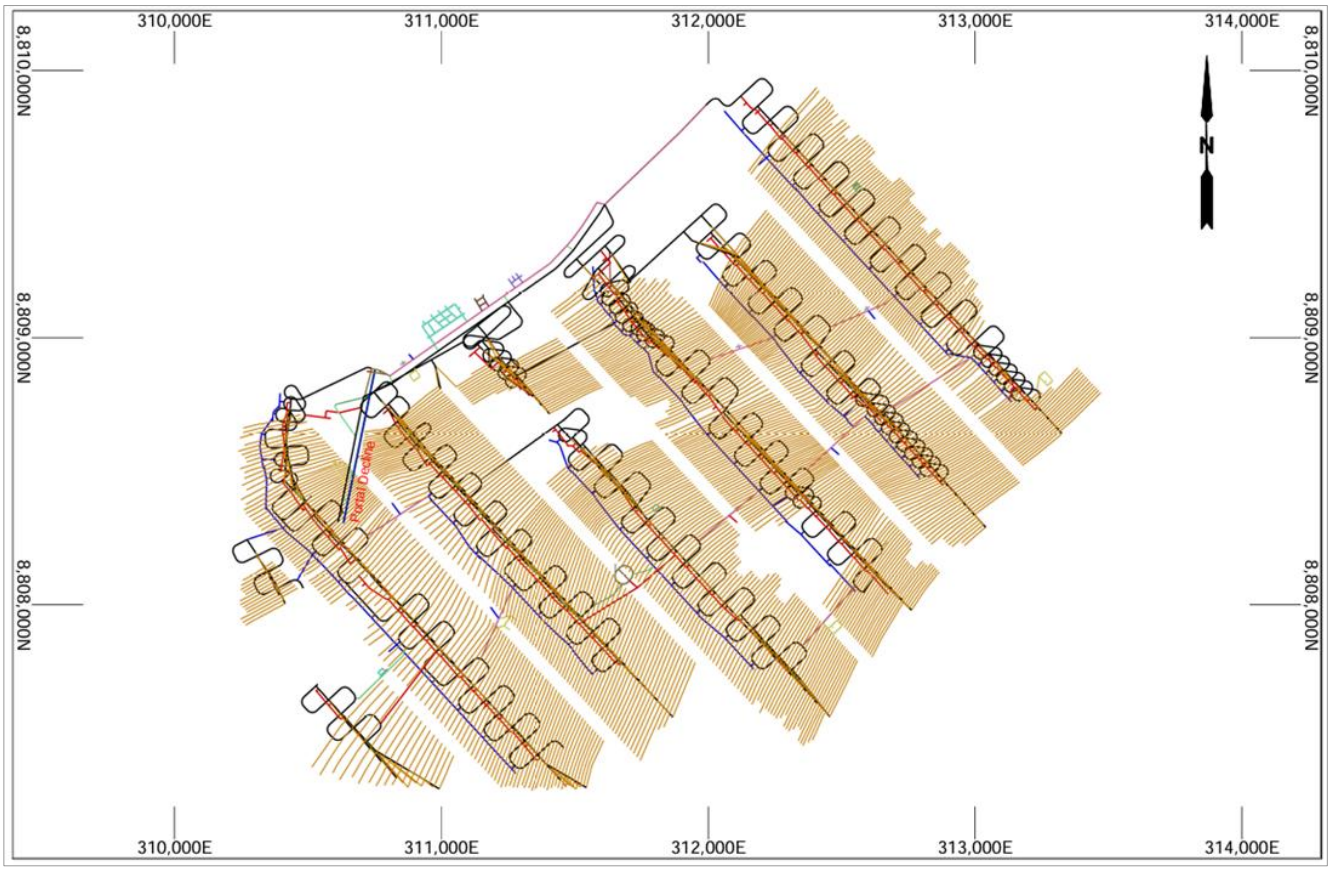
Table 16.45 shows the design assumptions for each of the tunnel activities.

Table 16.45 Excavation profiles

Description	Profile	Width (m)	Height (m)	Gradient / Dip
Major Access	Arched	6	6	Variable
Conveyor Decline	Arched	8	6	Variable
Connection Drive	Arched	5	5	Variable
Decline	Arched	6	6	1:7
Dewatering Station	Arched	6	6	Variable
Fresh Air Way	Arched	6	6	Variable
Fuel Bay	Arched	6	6	Variable
Haulage Drive	Arched	6	6	Flat
Major Service Droppers	Circle (Raisebore)	0.3 ⌀	-	90°
Rock Pass Access	Arched	6	6	Variable
Rock Pass	Circle (Raisebore)	5.0 ⌀	-	Variable
Ore Drive	Arched	4.5	4.5	Variable
Return Air Way	Arched	6	6	Variable
Refuge Bay	Arched	6	6	Variable
Slashed Access	Arched	4.5	4.5	Variable
Electrical Substation	Arched	6	6	Flat
Major Workshop	Arched	7	7	Flat
Return Air Raise	Circle (Raisebore)	6.0 ⌀	-	90°
Fresh Air Raise	Circle (Raisebore)	6.0 ⌀	-	90°

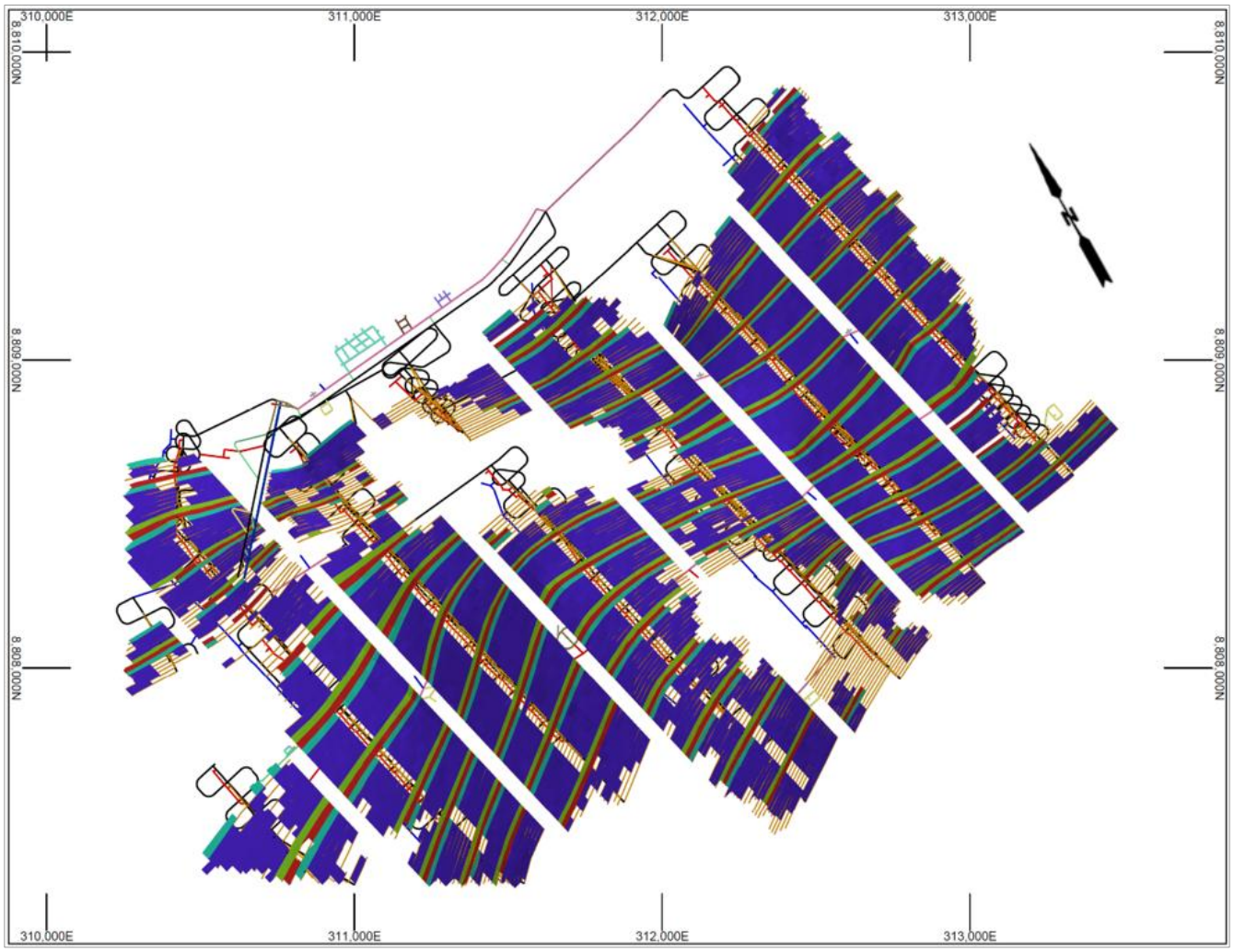
The design of the Kamoa 3 mining operation is illustrated in the following figures. (Figure 16.77, Figure 16.78, Figure 16.79) These images illustrate the multiple access declines and regional pillar configuration of the mining approach. The mining sequence combine production from multiple panels where each panel is mine in a bottom-up approach from a sill pillar that is also a crown pill for the panel below. On completion of mining from the panel, the crown pillar is extracted at a lower recovery factor. This approach enable multiple mining fronts across strike and along the down dip dimensions of the deposit.

Figure 16.77 Kamoa 3 plan view



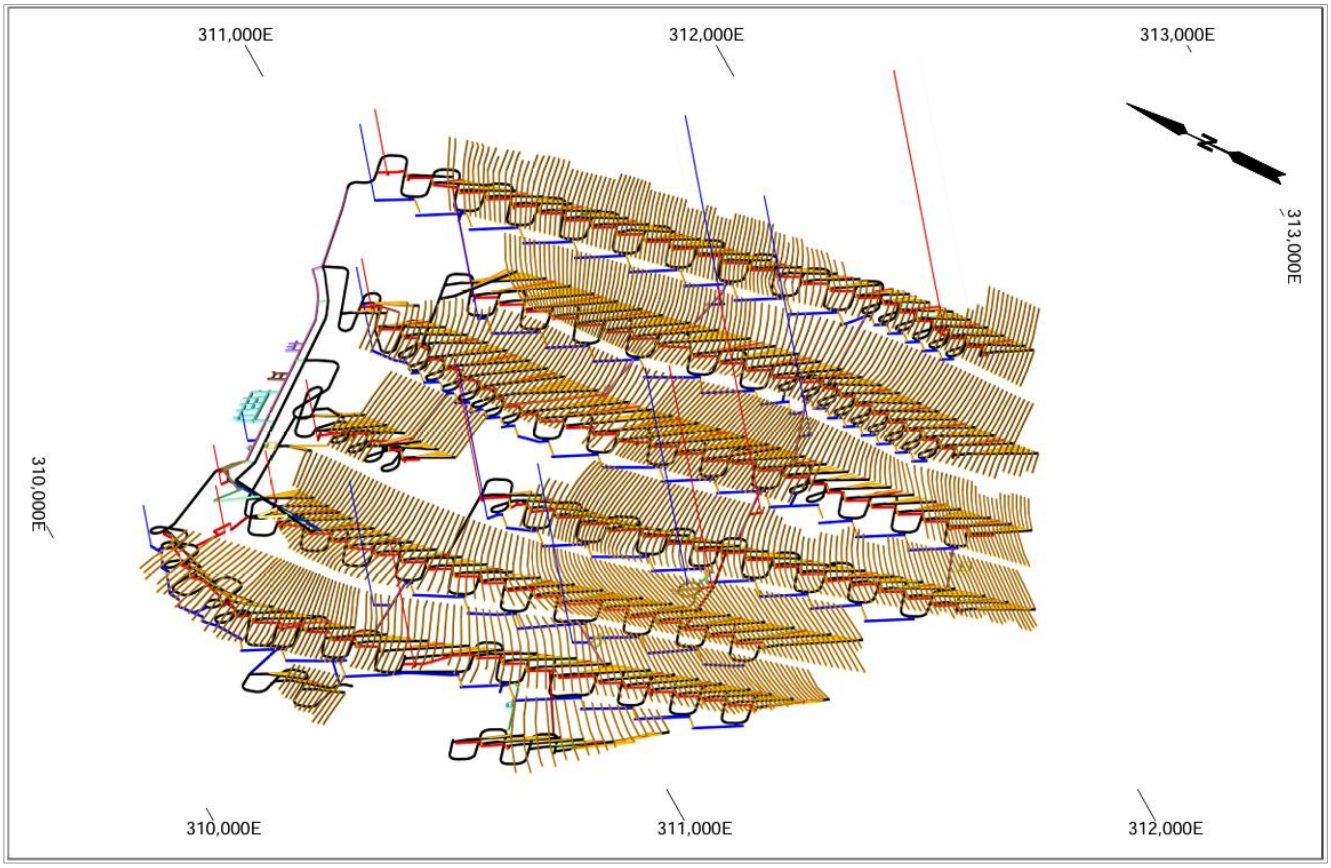
Source: AMC 2026

Figure 16.78 Kamoa 3 plan view with stopes



Source: AMC 2026

Figure 16.79 Kamoa 3 isometric view development only



Source: AMC 2026

16.9.5.2 Development mining and sequence profile

Table 16.46 shows the development advance by category and by year.

Table 16.46 LOM development

Year	Lateral development (m)	Vertical development (m)	Total development (m)
1	3400	213	3613
2	7083	356	7440
3	10991	846	11837
4	18472	1273	19745
5	21821	1989	23810
6	25393	1532	26925
7	26117	634	26751
8	26447	1673	28120
9	26566	381	26947
10	26571	977	27548
11	26571	1294	27864
12	25651	1377	27029
13	25576	1220	26796

Year	Lateral development (m)	Vertical development (m)	Total development (m)
14	22371	648	23019
15	17940	410	18350
16	18615	0	18615
17	15589	0	15589
18	15810	0	15810
19	16072	0	16072
20	11483	0	11483
21	9565	0	9565
22	5926	0	5926
23	3446	0	3446
24	3251	0	3251
25	2260	0	2260
26	3725	0	3725
27	2850	0	2850
28	1922	0	1922
29	2516	0	2516
30	2211	0	2211
31	2471	0	2471
32	2215	0	2215
33	1992	0	1992
34	2478	0	2478
35	1629	0	1629
36	2211	0	2211
37	1276	0	1276
38	459	0	459
39	246	0	246
Total	**441,189**	**14,822**	**456,011**

16.9.5.3 Infrastructure requirements

The Kamoa 3 mine will require a paste plant on surface, and a conveyor system from the upper underground levels to surface for transporting ore to the ROM and processing plant.

Materials handling

This low production rate mining method requires a high number of working areas to be active simultaneously to achieve a reasonable total production rate. The necessary dispersion of activities lends itself to a trucking method for materials handling.

Each mining domain will be accessed by a decline, and ore will be trucked to a central conveyor loading station. From there, all material will be conveyed to the surface and to the ROM.

Depending on future extensions to the mine, the option of a shaft hoist system may be considered later in the mine life as depth becomes an issue for the large trucking fleet.

Escapeways

The mine development design, shown in Figure 16.79 shows fresh air intakes in blue. These can double as escapeways and secondary egress. There are several connecting drives between declines which also allow for secondary egress via travel ways. These lateral connections are primarily required to reduce traffic congestion in the material handling system.

Ventilation and cooling requirement

Primary ventilation is enabled through several vertical shafts, established as required as the mine advances. Several primary exhaust and intake shafts will be required, with the primary exhausts each having a fan placed on top. Declines extend down-dip from the shaft locations, with parallel exhaust and intake airways extending along with it. Production areas have independent exhaust regulators that can be opened or closed to rapidly remove fumes and allow independent firing of levels. The production ore drives are fed via secondary ventilation from the decline access.

Water management

Dewatering will be undertaken by level sumps and decline sumps with secondary pumps reporting to several larger main pump stations. The primary pump stations connect directly to rising main pipes directly to the surface.

Power requirement

Power will be supplied underground via service dropper holes from the surface. These holes will be used to connect up to an underground substation and reticulation for each operational area.

Other

Compressed air will be provided by several underground compressor stations and receivers that will assist with reticulation.

Fixed infrastructure

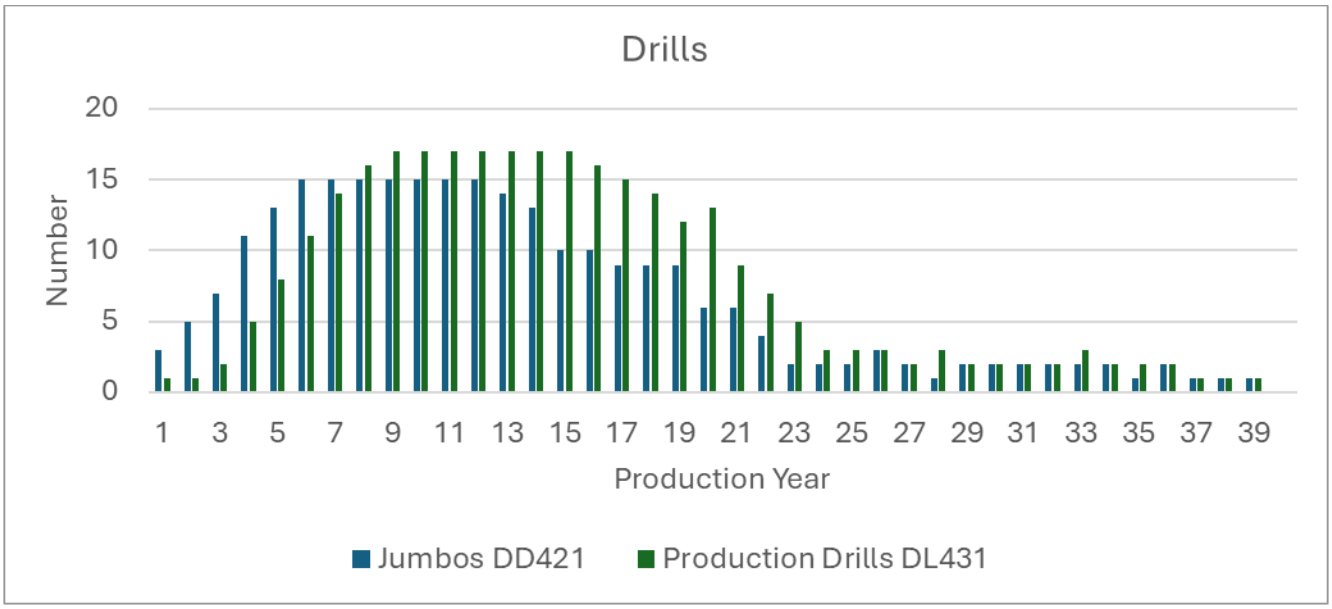
The design includes suggested locations for workshops, fuel bay, crew lunch room, and a suggested shaft location for a future materials handling review.

16.9.6 Kamoa 3 equipment requirements

16.9.6.1 Mobile equipment fleet

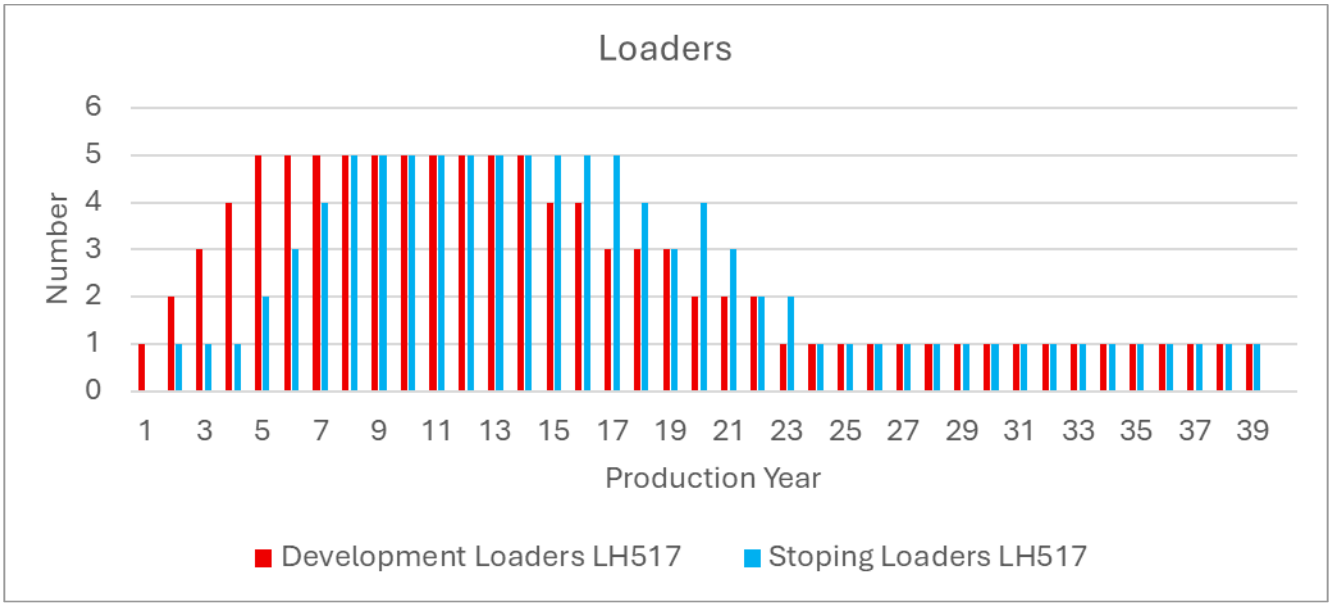
The following Figure 16.80, Figure 16.81 and Figure 16.82 show the planned equipment requirements for drills, loaders and trucks respectively.

Figure 16.80 Kamoa 3 drill equipment schedule



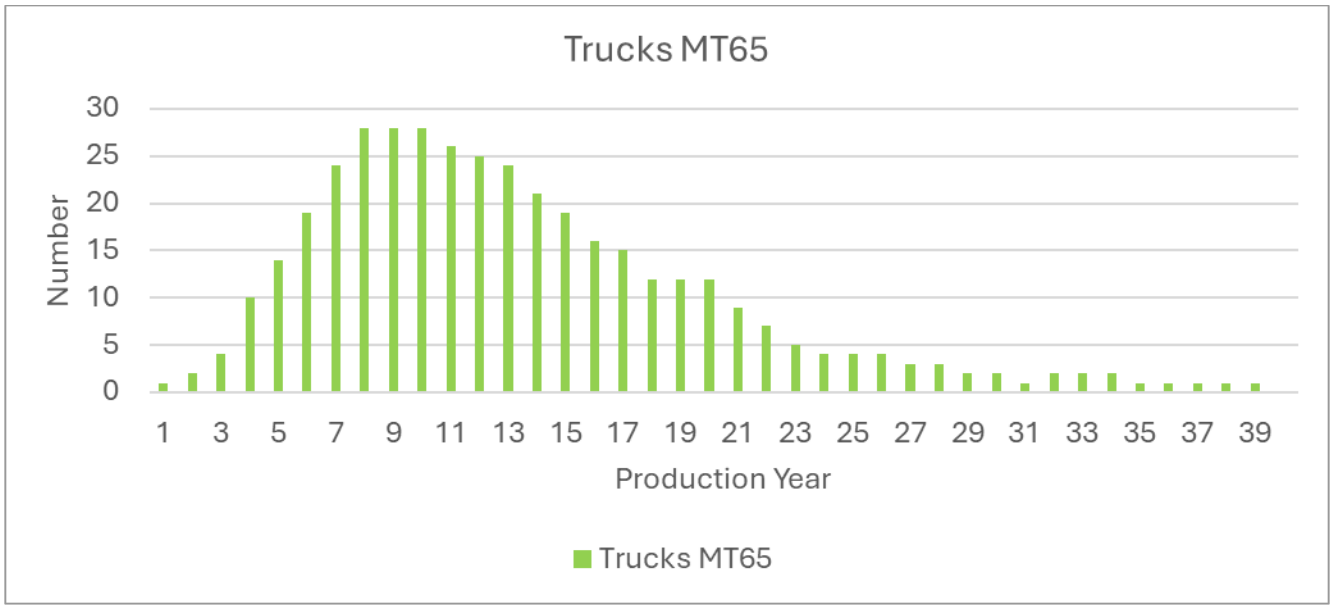
Source: AMC 2026

Figure 16.81 Kamoa 3 loader equipment schedule



Source: AMC 2026

Figure 16.82 Kamoa 3 truck equipment schedule



Source: AMC 2026

16.9.6.2 Fixed equipment

A conveyor and loading station will need to be installed from the second-year once ore production starts to increase. The loading station is located on the connecting haulage level, 130 m below surface.

16.9.7 Recommendations and next steps

The following items should be considered for the next phase of a Kamoa 3 study:

- A mining method better resembling traditional stoping should be considered for the stacked lens and thicker areas.
- The 7.5 m stoping height limitation should be reviewed in the context of additional numerical modelling and optimized pillar placement.

Capital efficiency of the design can be increased by focusing the extraction on thicker parts of the ore and treating peripherals to the thick zones as marginal ore.

- The 4.5 – 7 m marginal areas should include after assessment via a network flow algorithm. This will likely lead to a reduced footprint and much better capital efficiency.
- The current portal design cuts through some areas of higher grades, which could result in sterilization. The portal access location should be reviewed for a different location.
- Investigate connecting panels developed through the regional pillars. This may enable reducing the number of decline accesses but will add complexity to scheduling.
- Investigate placement of regional pillars according to orebody geometry.

An optimization study would benefit from a parallel materials handling trade-off investigation.

- This will help to determine whether a reduced footprint ore-body would benefit from a full-trucking system, or whether there is a more optimal location to place the conveyor.

- A shaft hoist option or conversion of a ventilation raise bore for materials handling could also be considered at depth.

Once the outcome of the above items has been determined, the mine design and schedule should be reviewed and more detailed design completed. Namely, the cut-and-fill accesses should be offset to avoid the creation of narrow floor pillars. Level design should be enhanced with the placement of drainage sumps and stockpiles, which are currently accounted for in numerical factors. Drive profiles should be reviewed on a case-by-case basis to match ore geometry.

16.10 Kamoa 4

16.10.1 Summary of relevant information

Kamoa 4

Kamoa 4 is characterized by a laterally extensive, thin, and sub-horizontal mineralization. The orebody extends approximately 5,500 m along strike (X direction) and 4,300 m across strike (Y direction). Mineralization occurs at relatively shallow depths, although a major north–south–trending fault displaces the western block downward by approximately 650 m, creating a significant depth differential across the deposit. The orebody dip ranges from 5° to 20°, with an average shallow inclination, reinforcing its tabular and stratiform geometry.

Given the thin, laterally continuous, and gently dipping nature of the mineralization, a room-and-pillar mining method has been selected for Kamoa 4.

Bonanza Areas

The Bonanza mineralization is structurally controlled and hosted within a fault zone where copper enrichment has occurred. Unlike Kamoa 4, the Bonanza domain exhibits a more complex geometry, comprising both steeply oriented, thicker mineralized zones and sub-horizontal areas with characteristics similar to those observed in Kamoa 4.

The Bonanza domain is subdivided into two principal areas:

- Bonanza Upper: approximately 1,500 m strike extent and 70 m vertical height, incorporating both vertical and sub-horizontal mineralized zones.
- Bonanza Lower: approximately 600 m strike extent and 50 m vertical height, also comprising vertical and sub-horizontal components.

The orebody has been structurally segmented by faulting, resulting in distinct upper and lower mineralized zones, with localized variations in dip and thickness. This structural complexity directly influences the mining method selection.

A dual mining approach has therefore been adopted:

- Bottom-up sublevel stoping has been selected for the steeply dipping, thicker vertical zones, where the geometry supports bulk extraction, longhole drilling, and gravity-assisted mucking.
- Room-and-pillar mining has been selected for the sub-horizontal areas, reflecting their tabular geometry and similarity to Kamoa 4. This method allows for controlled extraction while maintaining stability in flatter sections of the mineralization.

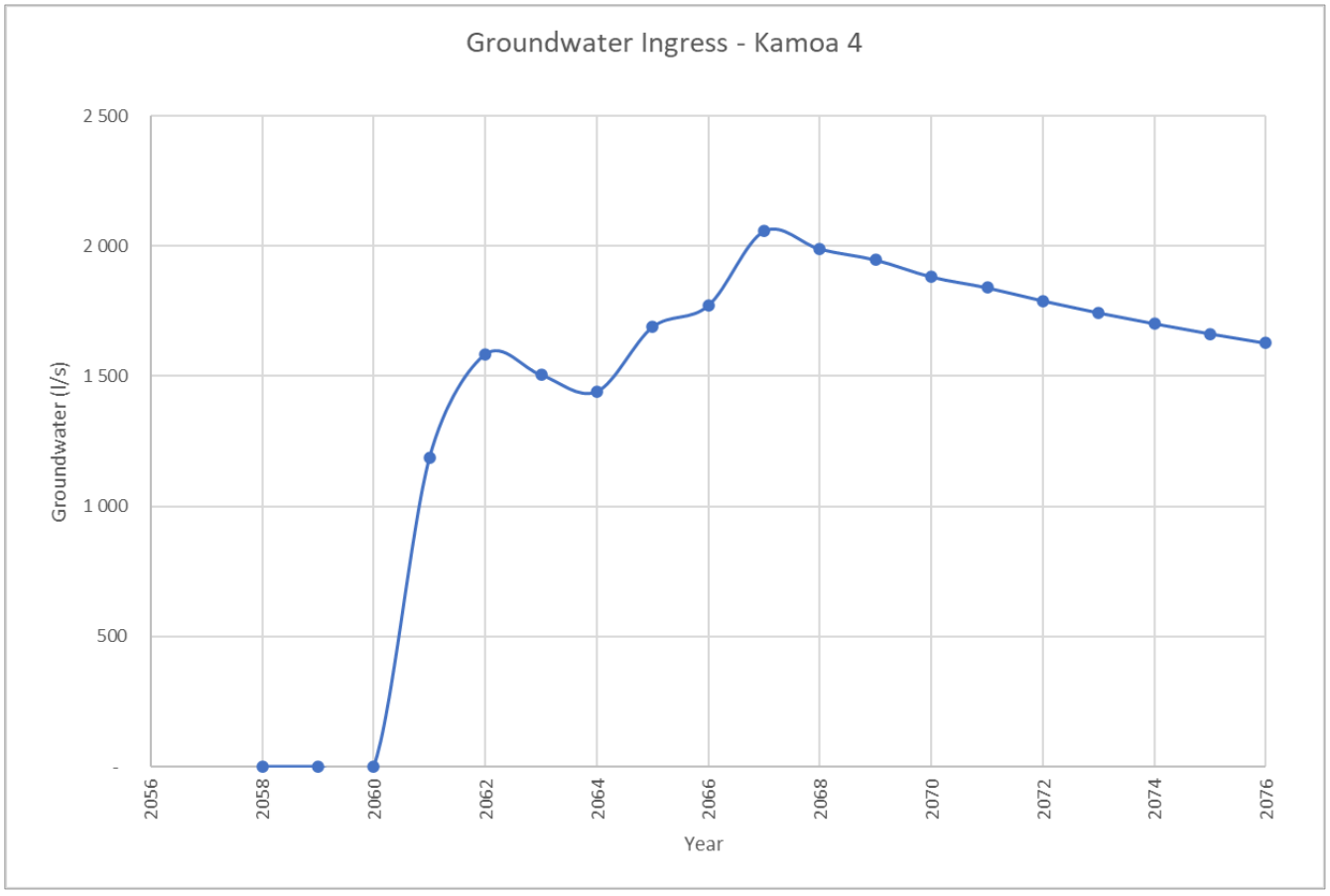
16.10.2 Dewatering for Kamoa 4

Kamoa 4 will have a main decline pumping system at decline bottom, which consists of a vertical dewatering dam which will pump out to surface via dewatering boreholes. The main decline pumping arrangement is capable of handling 1,000 l/s.

In addition to the main decline pumping system, two additional multistage dewatering pump stations will be located along the Kamoa 4 orebody – indicated by the green dots in Figure 16.84. Each multistage dewatering pump station will be capable of pumping 2,000 l/s to surface through dewatering boreholes.

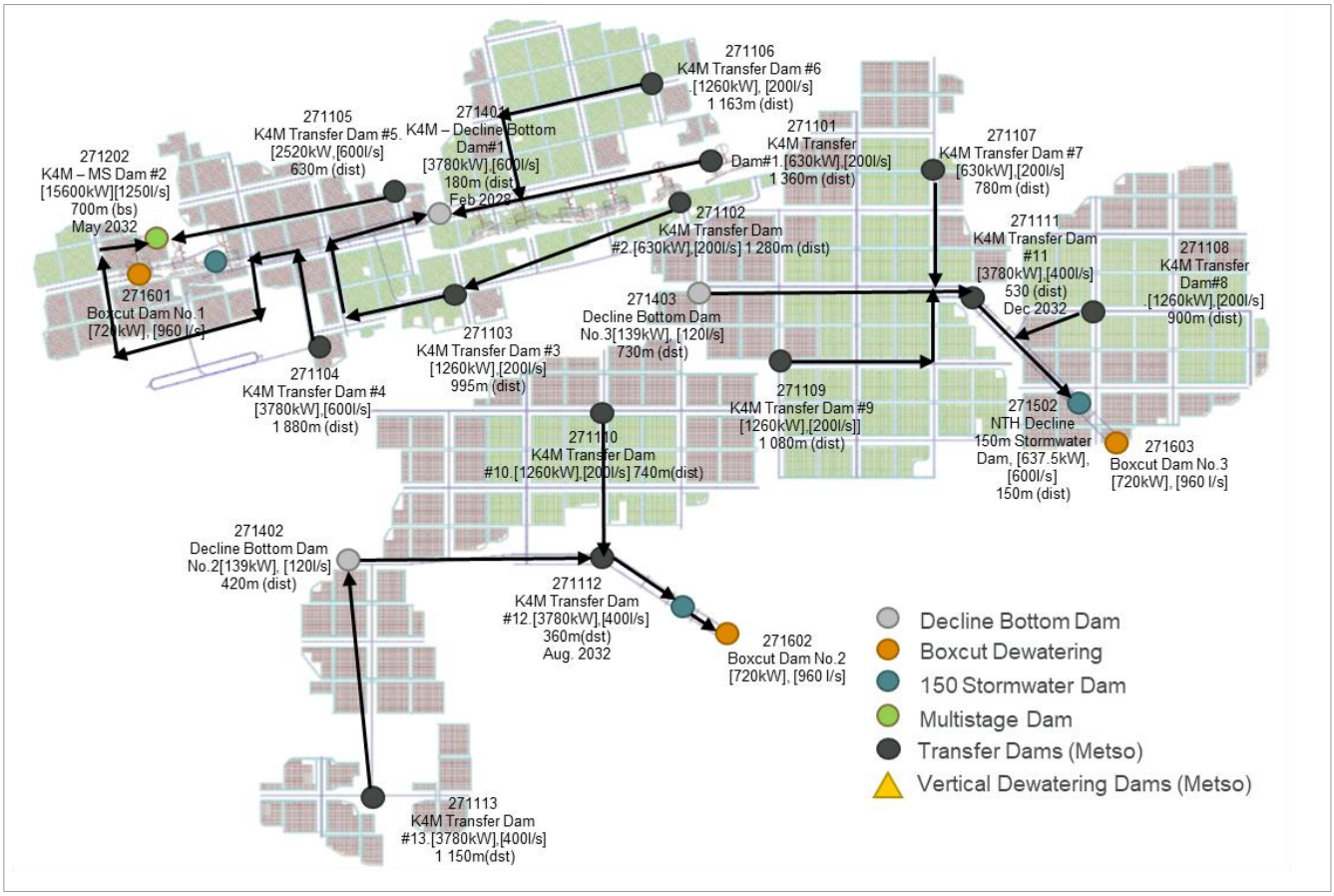
Kamoa 4 will have an expected peak water inflow rate of 2,100 l/s.

Figure 16.83 Kamoa 4 groundwater ingress



Source: WSP

Figure 16.84 Kamoa 4 Mine Dewatering Layout



Source: WSP 2026

16.10.3 Kamoa 4 Ventilation

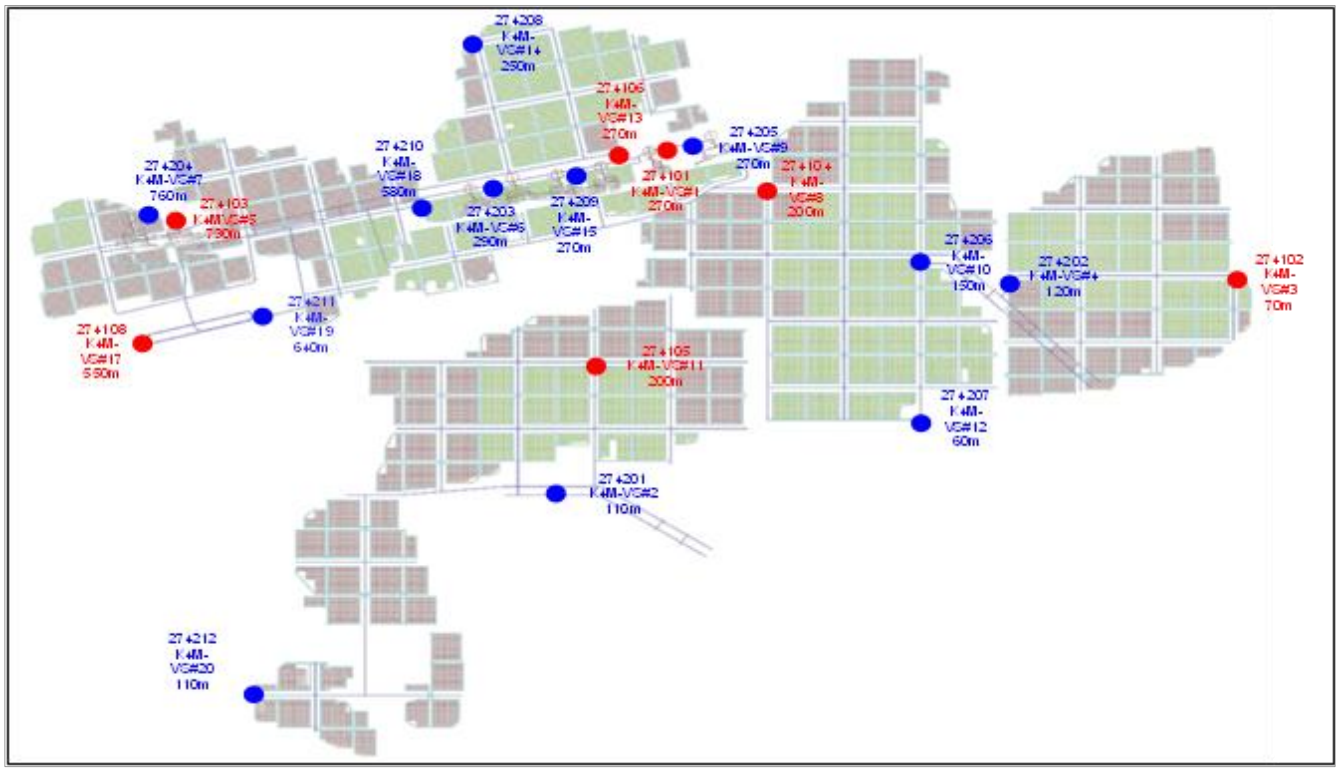
16.10.3.1 Kamoa 4 Ventilation strategy

The orebody will be accessed from the surface via the two twin decline systems, as shown in Figure 16.85 Fresh air will be supplied to the mine from the declines and eleven 6.0mØ intake ventilation shafts located across the orebody. Guided by ventilation controls, the air will flow through the perimeter drives to ventilate the development and stoping areas. Ore will be extracted using bord-and-pillar methods in shallow mining areas, while cut and fill mining will be implemented in the Bonanza sections.

After ventilating the production areas, the air will return to the surface through the upcast shafts

Figure 16.85 shows the mine layout, including the locations of the upcast and downcast shafts. The ventilation system is designed to provide fresh air from the declines and through intake ventilation shafts. The ventilation air will naturally flow through the drives connecting the extremities of the orebody. The model shows the primary ventilation requirements of 1,499 kg/s at the peak production rate of 7.2 Mtpa, and a maximum depth of 788 m.

Figure 16.85 Kamoa 4 Mine layout with ventilation shaft locations



DRA Global 11/11/2025.

16.10.4 Kamoa 4 production

The production plan for Kamoa 4 spans 17 years to produce 43 million tonnes of ore and 1.1 million tonnes of copper. The production profile is described below in Table 16.47. The primary focus for the development of Kamoa 4 will be the small, but high grade “Bonanza” zone. The remainder of the Kamoa 4 deposit will be mined as an integrated contributor to the overall KCSA mine plan.

Table 16.47 Kamoa 4 production profile

Future Operations	Ore Tonnes (Mt)	Grade (%Cu)	Cu Metal (Mt)	Peak Production (Mtpa)	Steady Production (Mtpa)	Life of Production (Yrs)
Kamoa 4	43.0	2.5	1.1	3.9	2.5	17

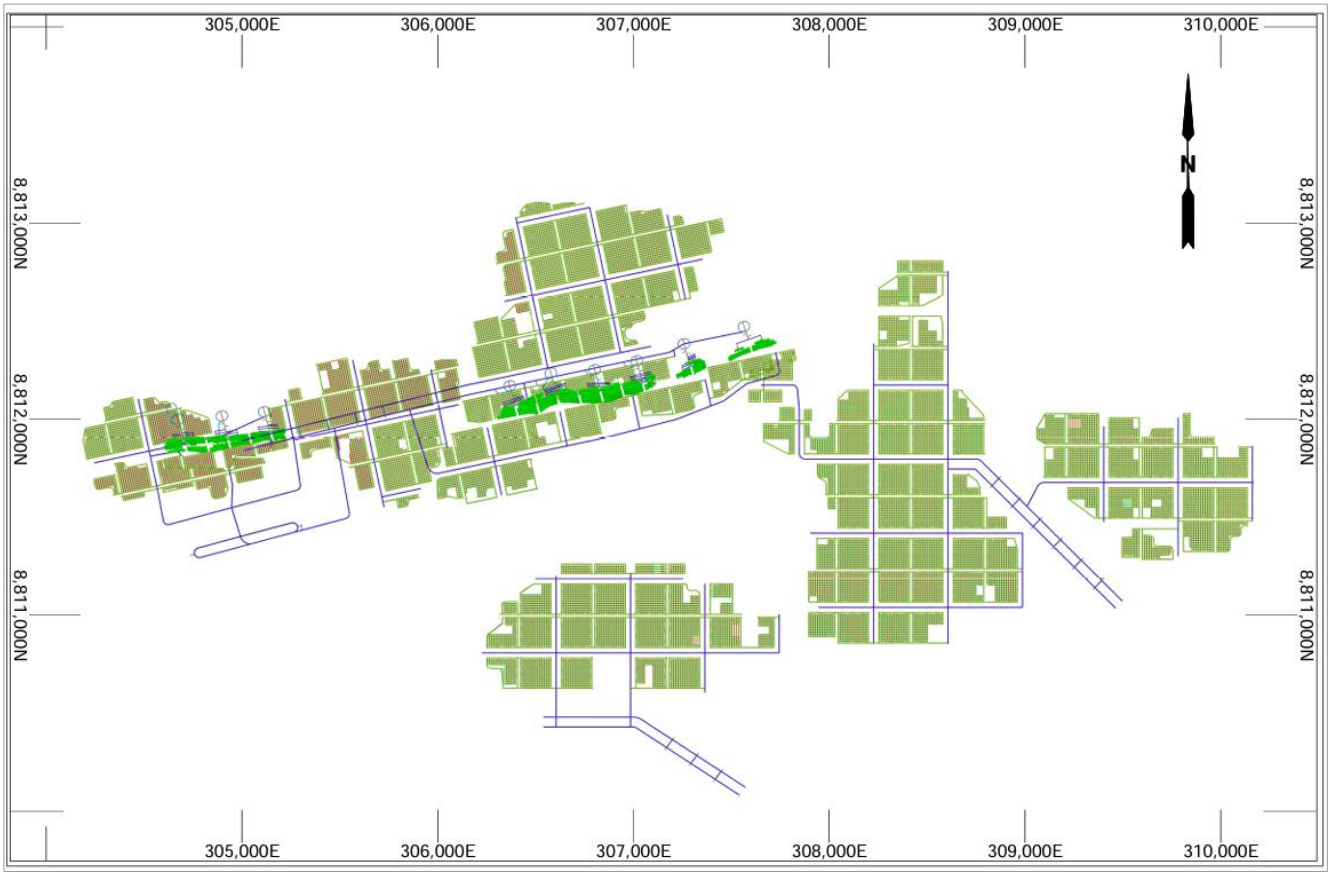
Equipment and task rates validated with the AMC SmartData™ benchmarking database.

Table 16.48 Machine productivity

Equipment or Task	Rate
Jumbo	130 m/month
Raisebore	2 m/day
Loader	1,200 t/day
Decline and Major Access	90 m/month
Lateral Development (Other)	50 m/month

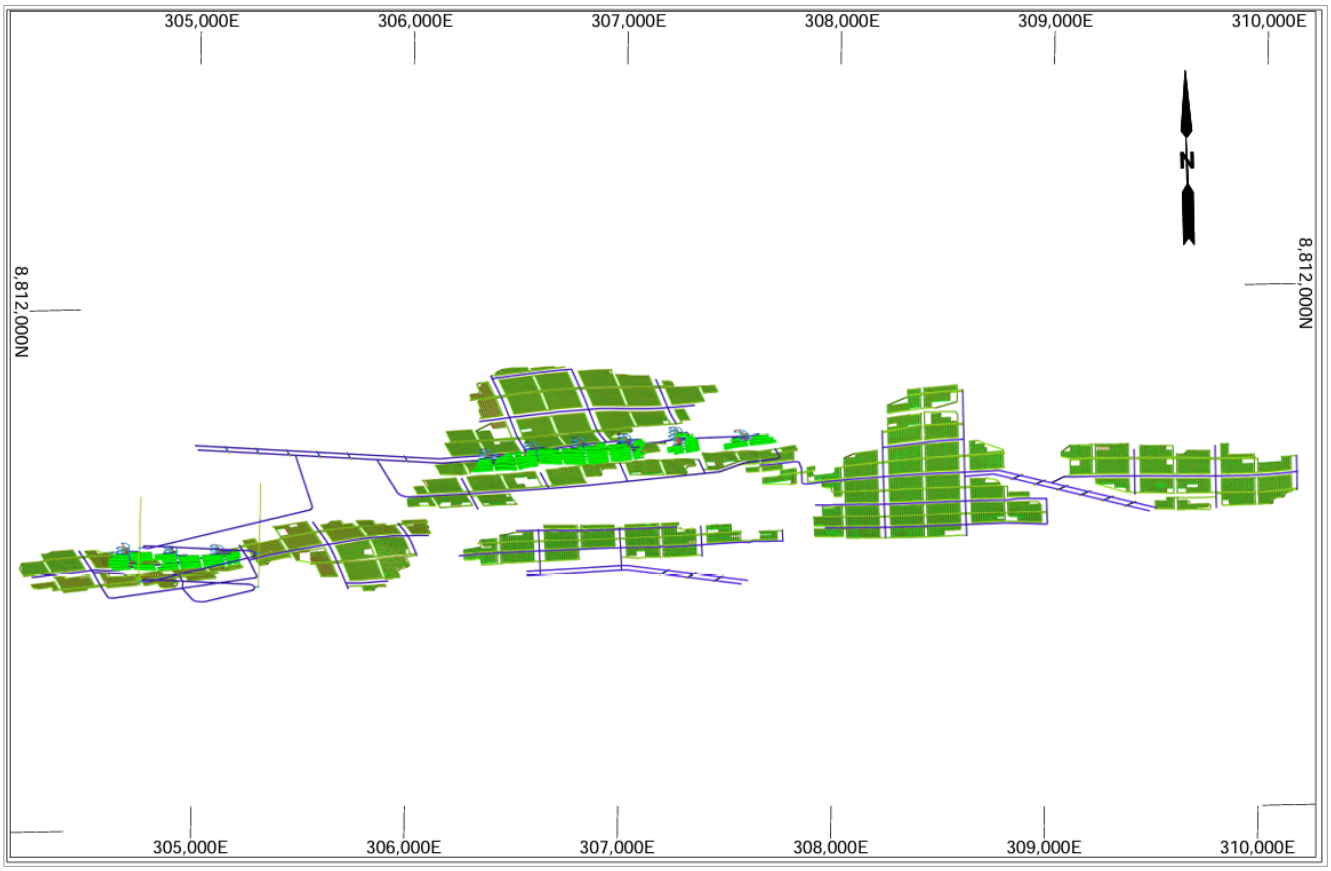
Material handling at Kamoa 4 is based on truck haulage via three twin declines, with ore and waste hauled directly from active mining areas to surface.

Figure 16.86 Kamoa 4 plan view



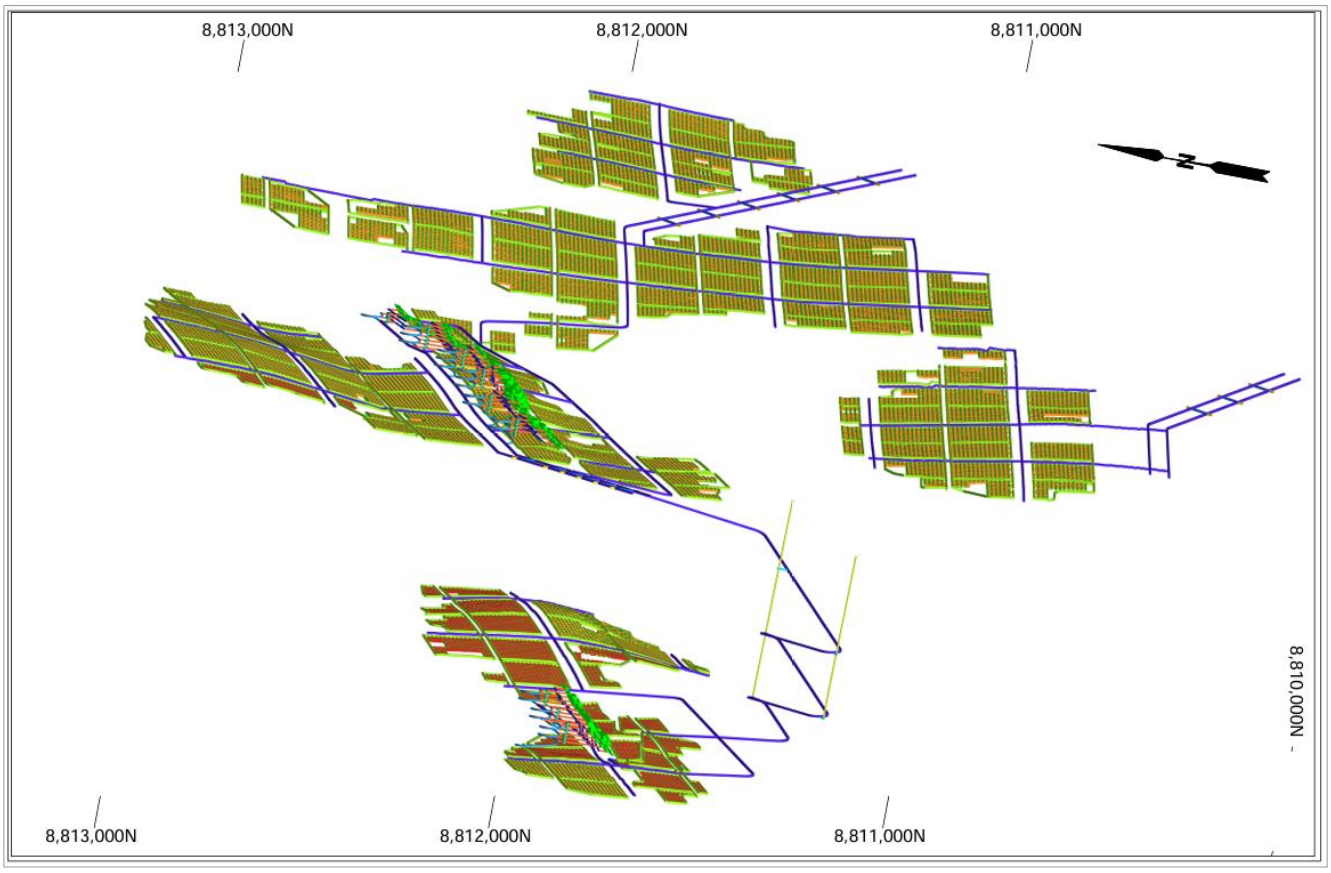
Source: AMC 2026

Figure 16.87 Kamoa 4 isometric view



Source: AMC 2026

Figure 16.88 Kamoa 4 isometric view



Source: AMC 2026

16.10.4.1 Production summary

Kamoa 4

Production at Kamoa 4 is based on a Room and Pillar mining method, selected to reflect the thin, laterally extensive, and sub-horizontal orebody geometry. The mining layout and extraction sequence have been designed to ensure stable production fronts, controlled ground conditions, and consistent ore delivery to the process plant.

The extraction sequence follows a structured and repeatable approach:

- Development of perimeter drives to establish panel limits and ventilation circuits.
- Advancement of panel entries to open production blocks.
- Lagged, retreat-style crosscut extraction progressing back toward the main access, maintaining geotechnical stability and operational flexibility.

Stope optimization was undertaken in two passes to balance productivity and economic viability:

- Primary optimization: 5 m (width) × 6 m (height) stope configuration, targeting improved equipment productivity and reduced unit costs.
- Secondary optimization: 5 m (width) × 4 m (height) configuration applied in areas where the larger geometry did not generate economic stopes during optimization.

These optimized mining dimensions were subsequently used as design guidelines for positioning crosscuts and panel entry drives, ensuring alignment between geological modelling, economic evaluation, and practical mine design.

Bonanza

Production at Bonanza is based on a combination of Sublevel Stoping and Room and Pillar, reflecting the variability in orebody geometry between the vertical / thicker zones and the more horizontal domains.

Sublevel Stoping Areas

In the steeper and thicker zones, Sublevel Stoping has been selected to maximize extraction efficiency while maintaining ground stability. The mining sequence is as follows:

- Development of access ramps to establish vertical connectivity.
- Establishment of crosscuts and ore drives along strike.
- Bottom-up extraction sequence, retreating from the outer edges toward central access points.
- Systematic backfilling of mined stopes to enable safe extraction of overlying stopes.

Key stope design parameters include:

- Minimum mining width: 3.0 m
- Stope height: 10 m
- Stope strike length: 10 m

Backfill is an operational requirement in these areas, enabling vertical sequencing and acceptable recovery ratios.

Room and Pillar Areas

In the more sub-horizontal zones, the mining approach mirrors that of Kamoa 4, utilizing a Room and Pillar method. The extraction sequence comprises:

- Perimeter and panel development.
- Panel entry advancement.
- Retreat mining via crosscuts back toward main access.

16.10.4.2 Modifying factors

The dilution and mining recovery factors applied to Kamoa 4 and Bonanza are summarized in the table below.

Table 16.49 Modifying factors for Kamoa 4 and Bonanza

Activity	Dilution factor	Mining recovery factor
Horizontal development	1.00	1.00
Vertical development	1.00	1.00
Primary stope	1.00	1.00
Secondary stope	1.00	1.00

No additional modifying factors for dilution or mining recovery have been applied beyond those reflected in the mine design. This approach is considered appropriate for this level of study as dilution and recovery assumptions are inherently incorporated within the stope and development geometries defined during the design process.

16.10.5 Kamoa 4 development and infrastructure

Infrastructure at Kamoa 4 is based on a conventional decline-accessed underground layout. Primary access is provided via three twin-decline portals, with ore and waste hauled by truck directly to surface.

The lower areas of Kamoa 4 and Bonanza are accessed through a decline originating from the western surface twin decline, providing integrated access, ventilation continuity, and operational flexibility across both domains.

16.10.5.1 Basis of Design

Kamoa 4 excavation dimensions are described in Table 16.50

Table 16.50 Excavation profiles

Description	Profile	Width (m)	Height (m)	Gradient / Dip
Major Access	Arched	6	6	Variable
Bypass	Arched	4	4	Variable
Connection Drive	Arched	6	6	Flat
Decline	Arched	6	6	Variable
Fresh Air Way	Arched	6	6	Variable
Ore Pass Access	Arched	5	5	Variable
Ore Pass	Circle (Raisebore)	2.5 ⌀	-	90°
Ore Drive	Arched	5	5	Variable
Panel Entry Drive	Arched	5	6	Variable
Panel Entry Drive 4	Arched	5	4	Variable
Perimeter Drive	Arched	6	6	Variable
Stockpile	Arched	6	6	Flat
Cross Cut	Arched	5	6	Variable
Cross Cut 4	Arched	5	4	Variable
Cross Cut BZ	Arched	4	4	Variable
Return Air Raise	Circle (Raisebore)	4.5 ⌀	-	90°
Fresh Air Raise	Circle (Raisebore)	4.5 ⌀	-	90°

16.10.5.2 Development mining and sequence profile

LOM development profile for Kamoa 4 is shown in Table 16.51

Table 16.51 LOM development

Year	Lateral development (m)	Vertical development (m)	Total development (m)
1	9368	0	9368
2	13835	416	14251
3	21048	361	21409
4	17174	167	17341
5	27352	0	27352
6	30183	0	30183
7	29156	0	29156
8	37025	0	37025
9	94244	0	94244
10	81476	0	81476
11	64492	0	64492
12	47140	0	47140
13	39851	0	39851
14	19911	0	19911
Total	**532,255**	**944**	**533,199**

16.10.6 Recommendations and next steps

Before the next stage of study, additional data collection of the Bonanza area is required for geotechnical and grade data and to confirm mine design assessments made in this work. In the next stage of study, an assessment of alternative mining method options should be undertaken. This evaluation should compare technical and economic performance across different geological domains, considering factors such as recovery, dilution, sequencing flexibility, geotechnical risk, productivity, and overall project value.

Dilution and mining recovery assumptions should be refined for each mining method. These estimates should be informed by detailed design parameters, updated geotechnical information, and benchmarking against comparable operations

16.11 Kamoa 5

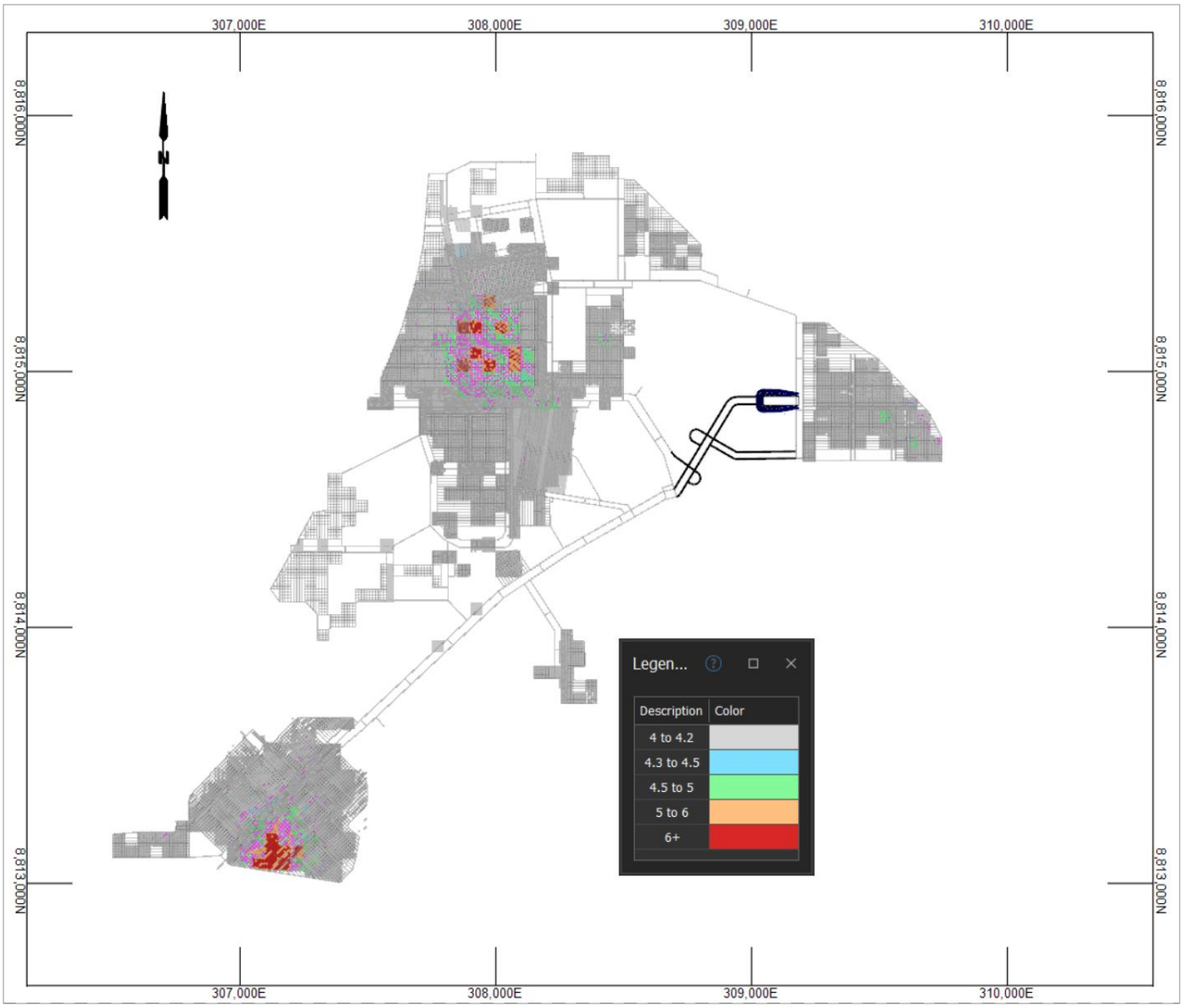
16.11.1 Summary of relevant information

The Kamoa 5 deposit forms part of the broader Kamoa mining complex and is characterized by a laterally extensive, shallow-dipping stratiform copper orebody. The orebody vertical thickness is predominantly narrow, with most of the mineralization ranging between approximately 4.0 m and 4.2 m, and localized thickening up to approximately 6 m. The dip is generally shallow at 2–5°, with isolated steeper zones locally reaching up to approximately 20°. The deposit occurs at relatively shallow depths, ranging from approximately 60 m to 220 m below surface.

A major geological structure, WS1, intersects the Kamoa 5 orebody. As a precautionary design measure, a minimum 50 m stand-off from this fault zone has been incorporated into the mine layout to reduce potential geotechnical and operational risk during mining. Outside the influence of this structure, the orebody

exhibits broadly consistent geometry and continuity, supporting the application of laterally repetitive mining panels and standardized development layouts.

Figure 16.89 Kamoa 5 average mining height in metres



Source: AMC2026

The selected mining method for Kamoa 5 is predominantly Room-and-Pillar (R&P) mining, with limited application of cut-and-fill or slashing in isolated steeper or locally thickened zones. This selection is directly driven by the shallow dip, relatively uniform but thin orebody thickness, and shallow depth below surface. (See section 16.4.3)

While the R&P mining method is development-intensive and relatively high unit cost, and typically achieves lower overall recovery compared to conventional stoping methods, it offers a high degree of selectivity and consistent operating performance. Provided the orebody geometry does not change significantly with additional drilling and considering the thin nature of the Kamoa 5 orebody together with the limited

likelihood of paste availability, R&P mining has been selected as the most appropriate and practical mining method for this deposit.

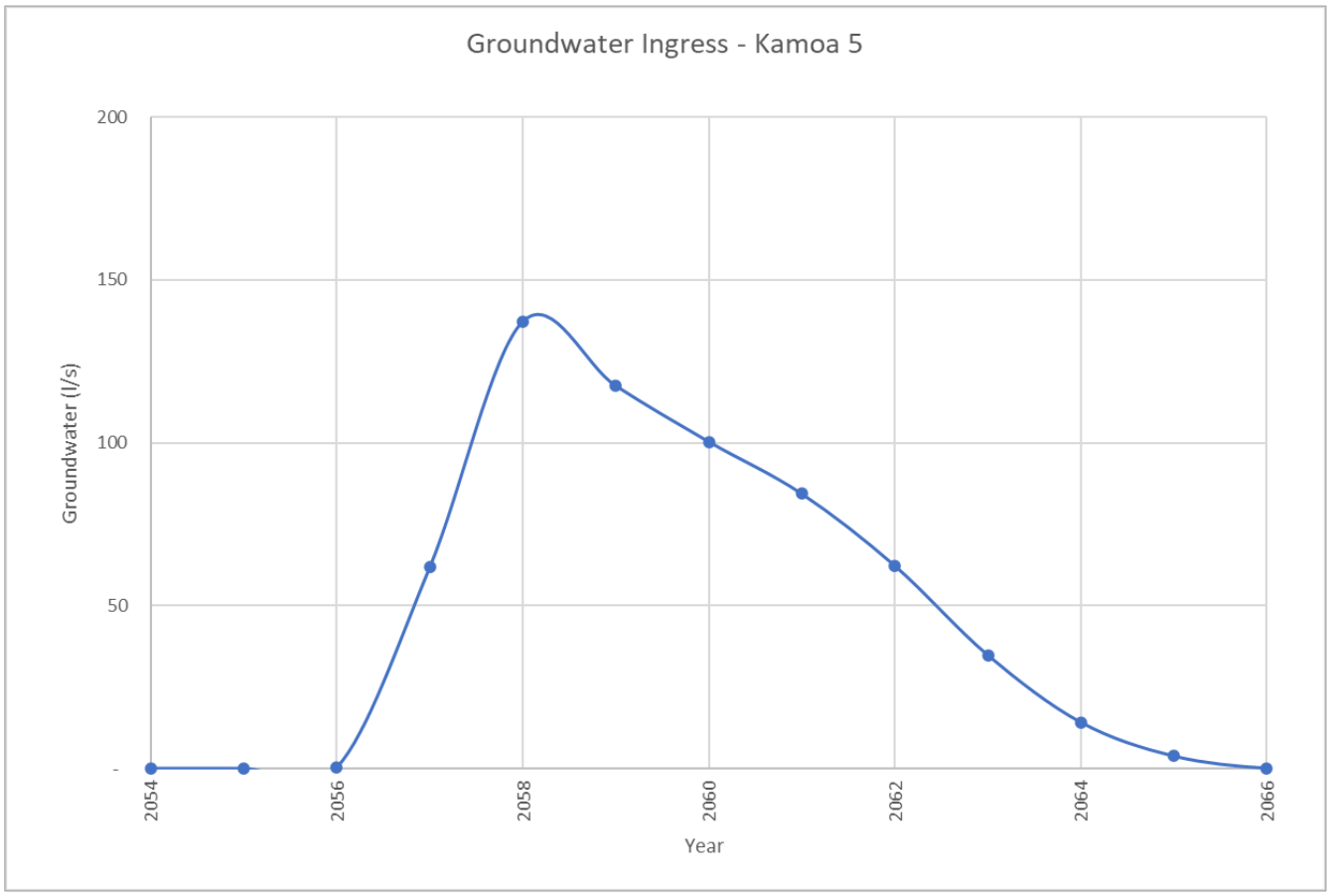
Chain pillar dimensions are governed by the extraction drive height, which is 4 m in the current design. In accordance with Beck Engineering guidance, a 1:2 height-to-width ratio is applied, resulting in chain pillars of 8 m width to provide regional stability between production panels. Regional pillars are scheduled for extraction towards the end of the production life, subject to prevailing ground conditions and operational constraints, with the mine plan assuming an overall 30% extraction ratio. Further detail on pillar design rationale and recovery assumptions is provided in the Beck Engineering guidance.

16.11.2 Dewatering for Kamoa 5

Kamoa 5 will have a main decline pumping system located at the bottom of the decline. This system consists of a vertical dewatering dam that pumps water to the surface via dewatering boreholes. The main decline pumping arrangement is designed to handle 400 l/s.

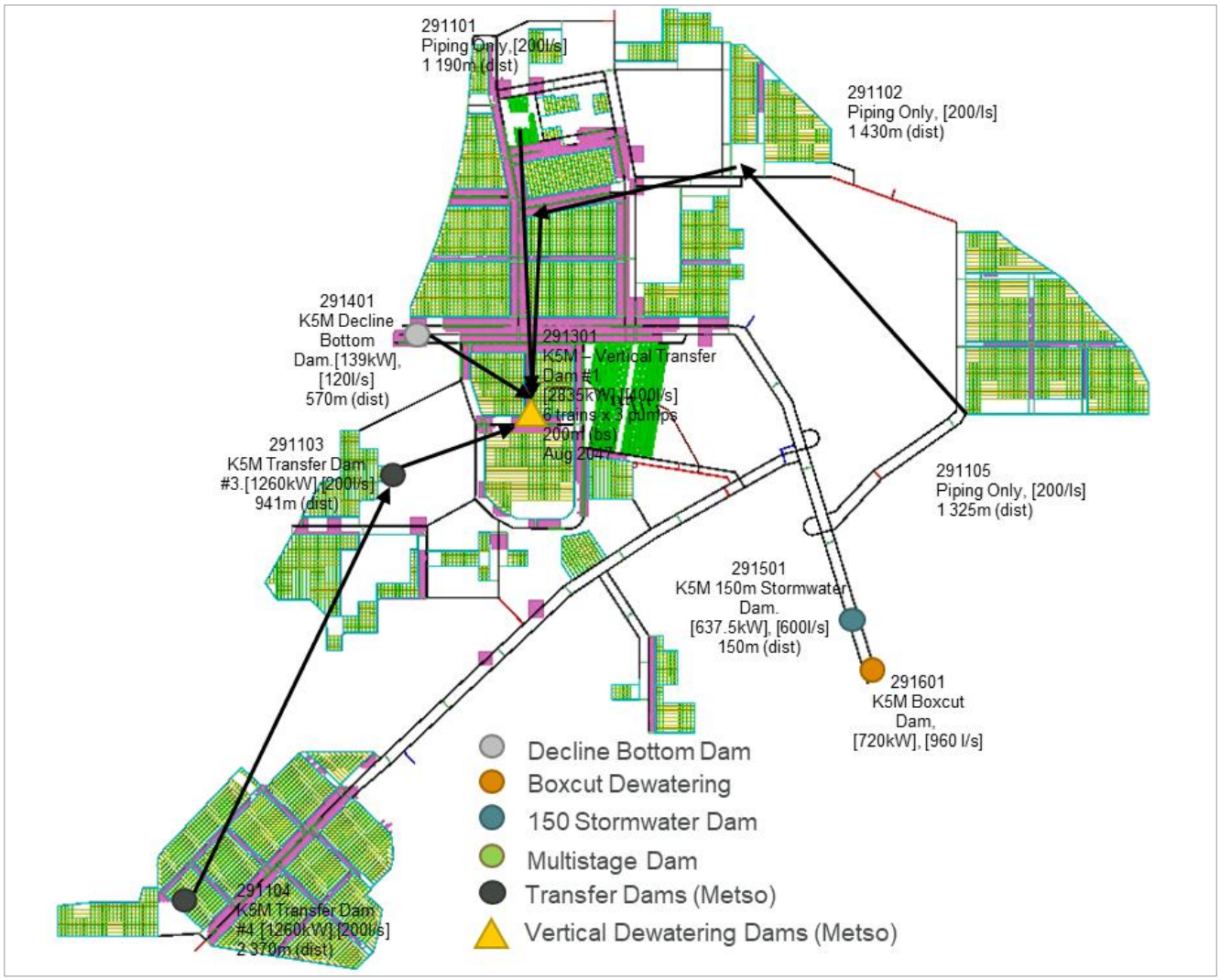
Kamoa 5 will have an expected peak water inflow rate of 140 l/s.

Figure 16.90 Kamoa 5 groundwater ingress



Source: WSP 2026

Figure 16.91. Kamoa 5 Mine Dewatering Layout



Source: WSP 2026

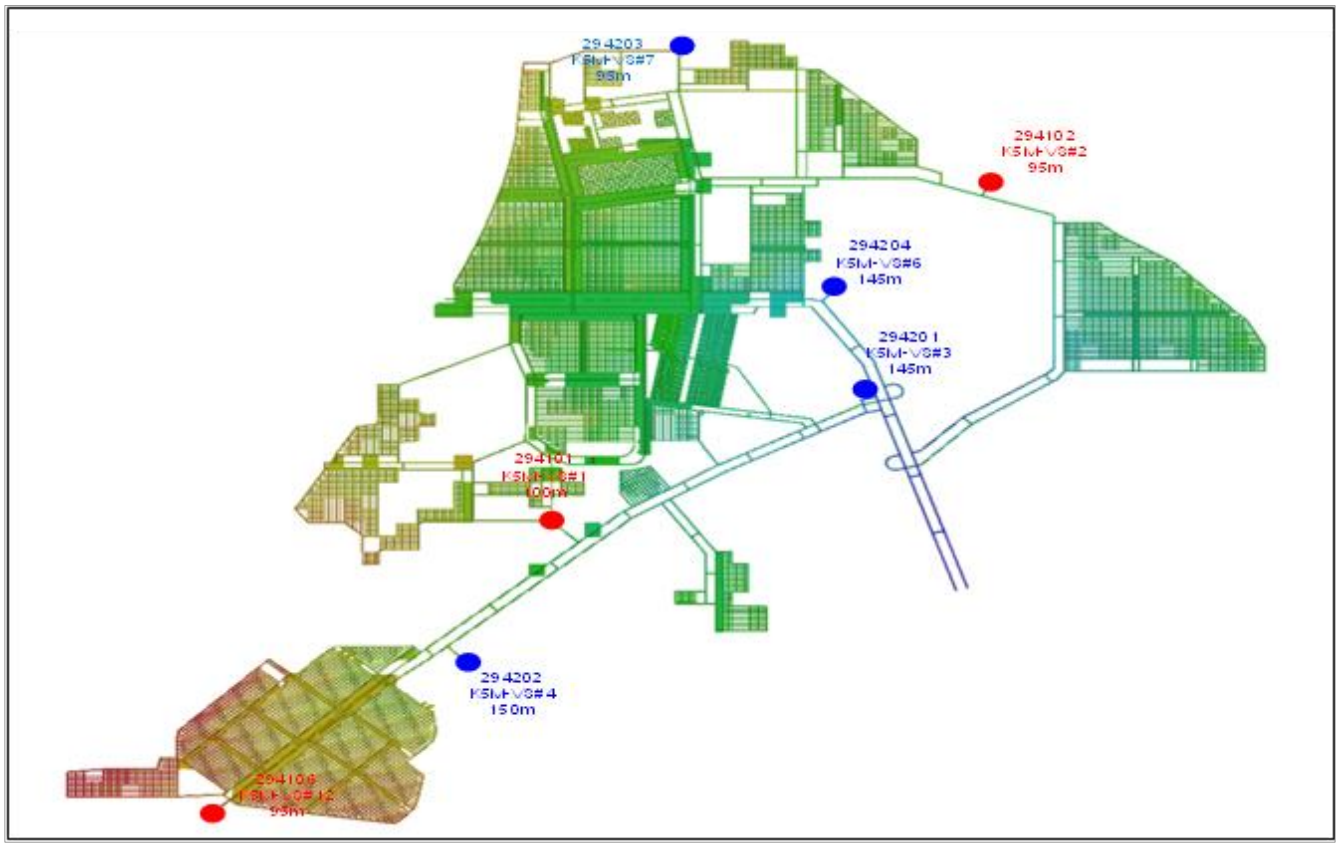
16.11.3 Kamoa 5 Ventilation

The Kamoa 5 orebody will be accessed from the surface via the twin decline system, as shown in Figure 16.92. Fresh air will be supplied to the mine from the declines and four 6.0mØ intake ventilation shafts located across the orebody. Guided by ventilation controls, the air will flow through the perimeter drives across the orebody to ventilate the development and stoping areas. Ore will be extracted by means of R&P due to shallow depth of the orebody.

Figure 16.92shows the mine layout with the location of ventilation upcast and downcast shafts. Air will return from south perimeter drives via three upcast shafts to surface. The overall ventilation quantity for this mine is determined by the need to dilute diesel fumes, remove heat (distribute cooling) and dilute blasting fumes during re-entry. It must be noted that the overriding factor here is to dilute diesel fumes.

The model shows the primary ventilation requirements of 971 kg/s at the peak production rate of 4.3 Mtpa, and a maximum depth of 252 m.

Figure 16.92 Kamoa 5 Mine layout with ventilation shaft locations



Source: DRA Global 11/11/2025.

16.11.4 Kamoa 5 production

The production plan for Kamoa 5 spans 17 years to produce 9 million tonnes of ore and 0.2 million tonnes of copper. The production profile is described below in Table 16.52.

Table 16.52 Kamoa 5 production profile

Future Operations	Ore Tonnes (Mt)	Grade (%Cu)	Cu Metal (Mt)	Peak Production (Mtpa)	Steady Production (Mtpa)	Life of Production (Yrs)
Kamoa 5	8.6	2.7	0.2	1.5	0.7	12

Equipment and task rates applied to the validated with the AMC SmartData™ benchmarking database.

Table 16.53 Kamoa 5 equipment and task rate

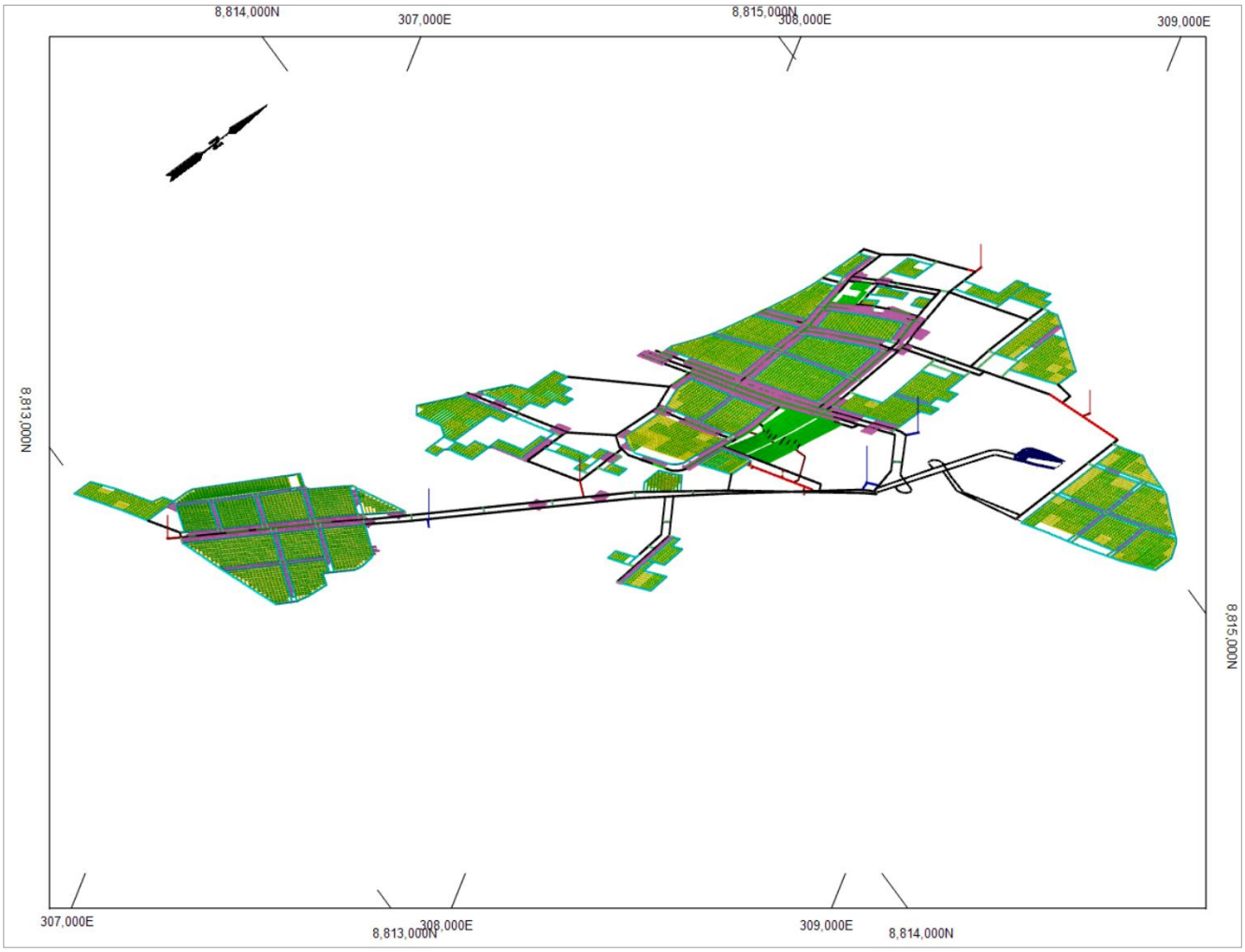
Equipment or task	Rate
Jumbo	130 m/month
Production drill	150 m/day
Raisebore	2 m/day
Loader	800 t/day
Cable bolter	150 m/day
Decline and major access	90 m/month
Lateral development (other)	50 m/month
Intersection cable	36 m

Material handling at Kamoa 5 is based on truck haulage via twin declines, with ore and waste hauled directly from active mining areas to surface. With minimal material movement in the early years, increasing to a peak production rate of approximately 1–1.5 Mtpa during the main production period before declining toward end of mine life.

Figure 16.93 Kamoa 5 plan view

Source: AMC2026

Figure 16.94 Kamoa 5 isometric view



Source: AMC2026

16.11.5 Production summary

Production at Kamoa 5 is based on a predominantly R&P mining approach, with minimal stoping or slashing planned, reflecting the thin orebody geometry and limited paste availability. Mining follows a structured sequence comprising perimeter drive development, panel entry advancement, and lagged, retreating crosscut extraction back toward main access, providing consistent production fronts.

Backfill is not relied upon as a core component of the mining method. Consequently, approximately 900 kt at an average grade of 2.2% Cu is expected to be sterilized in the absence of paste, in addition to 261 kt at an average grade of 1.68% Cu sterilized within the 70 m crown pillar.

16.11.5.1 Modifying factors

Table 16.54 Kamoa 5 modifying factors

Activity	Dilution factor	Mining recovery factor
Horizontal development	1.1	0.95
Vertical development	1.0	1.0
Primary stope	1.1	0.95
Secondary stope	1.15	0.9

16.11.5.2 Kamoa 5 development and infrastructure

Infrastructure at Kamoa 5 is based on a conventional, decline-accessed underground mine layout that supports the R&P mining method. Primary access to the mine is provided via a twin-decline portal, with ore and waste hauled by truck directly from underground working areas to surface.

The ventilation system is configured into multiple independent ventilation districts to support concurrent mining activities. Fresh air is supplied through three surface intake shafts supplemented by portal intake, while exhaust air is returned to surface via four dedicated exhaust shafts. Perimeter drives are developed early in the mine life to establish through-ventilation and support subsequent panel development.

Mine dewatering is managed through drain holes that collect water to pump stations located at the mine bottom, from where water is pumped to surface. Localized water management infrastructure is incorporated within individual mining domains to manage inflows as mining advances laterally.

Power and services are reticulated along the main access and declines, with infrastructure extended progressively in line with mine development.

16.11.5.3 Basis of design

The basis for the design of the Kamoa 5 operation is described in Table 16.55

Table 16.55 Excavation profiles

Description	Profile	Width (m)	Height (m)	Gradient / Dip
Major access	Arched	6	6	Variable
Auxiliary drive	Arched	5	5	Variable
Connection drive	Arched	6	6	Variable
Decline	Arched	6	6	1:7 UP / DOWN
Fresh air way	Arched	6	6	1:50 UP
Ore drive	Square	5	5	Variable
Panel entry drive	Arched	5	4	Variable
Perimeter drive	Arched	5.5	5.5	Variable
Return air way	Arched	6	6	1:50 UP
UG return air raise	Square	2.5	2.5	90°
Surface ventilation shaft	Circle	3.5Ø		90°
Crosscut	Arched	5	4	Variable

16.11.5.4 Development mining and sequence profile

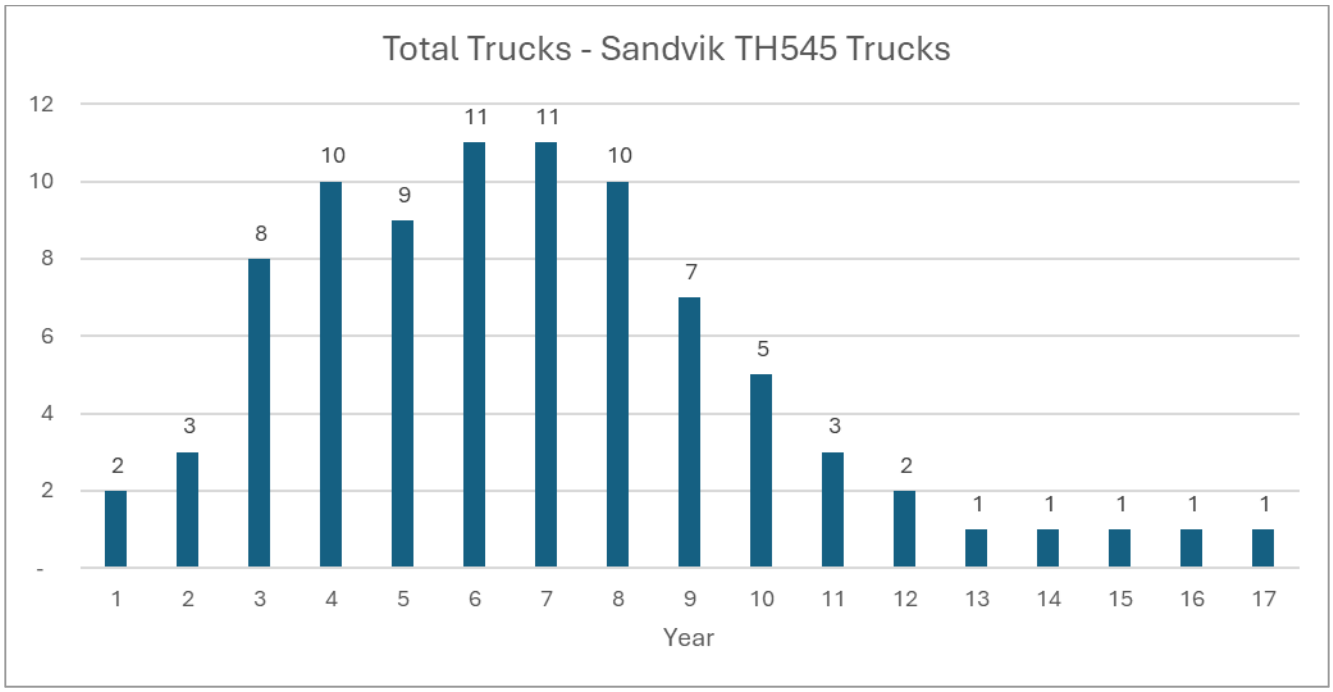
Development activities at Kamoa 5 are dominated by horizontal excavation consistent with the application of the room-and-pillar mining method. Of the approximately 193 km of total planned development, more than 99 % comprises lateral development, with only 646 m of vertical development required, primarily for early-stage ventilation infrastructure. Development intensity increases significantly during the ramp-up period, with the highest annual advance occurring between Years 4 and 9 as operating development activities overlap with the expansion of capital development into additional mining areas. This results in a front-loaded development profile characterized by a higher proportion of capital infrastructure established early in the mine life. As mining areas mature, development progressively transitions to predominantly operating requirements, followed by a steady decline in total development demand toward the end of mine life as production areas are depleted.

16.11.5.5 Mobile equipment fleet

Please refer to the below figures for planned equipment requirements throughout the mine life.

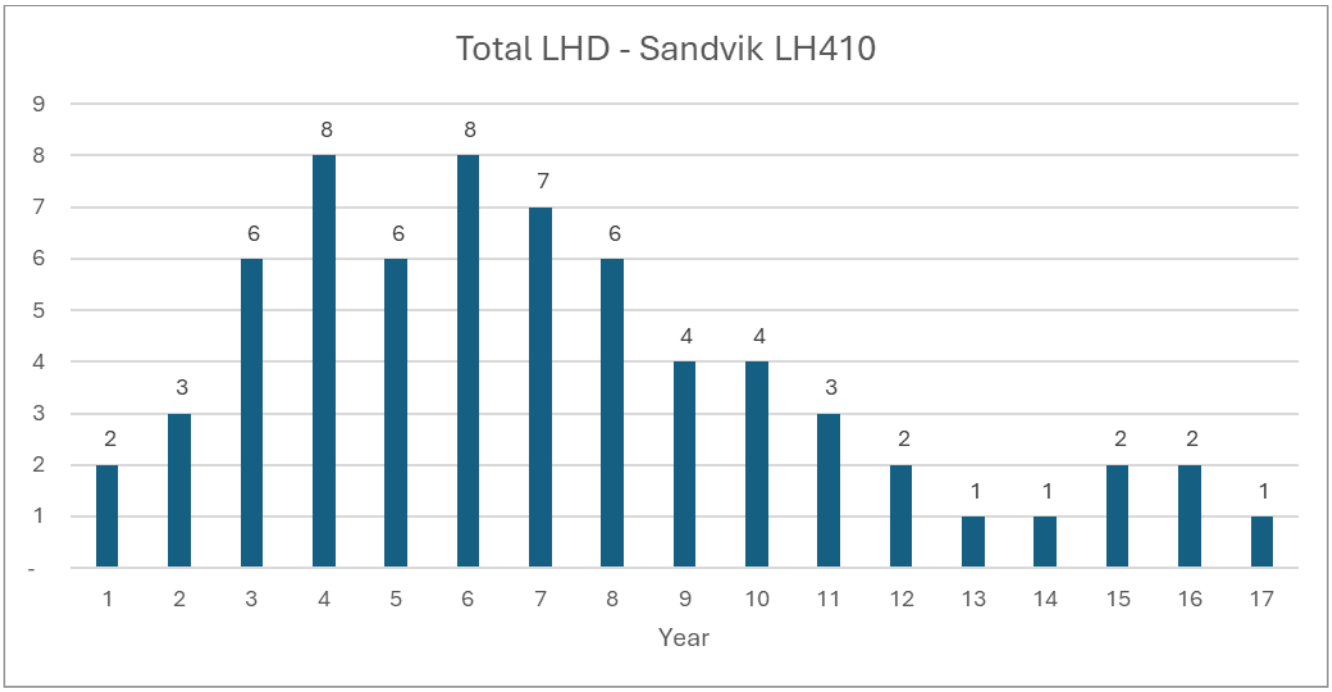
Truck and LHD requirements increase steadily during the early years as R&P production areas are established, peaking during the mid-mine life when production and development activity are at their highest. Notably the truck and LHD unit selection assumes smaller size units than at the other Kamoa mines (specifically Sandvik TH545 and Sandvik LH410) to suit the production area heights. Drilling requirements are dominated by development face drilling, reflecting the development-intensive nature of the R&P mining method. As mining activity reduces toward the end of mine life, fleet numbers decrease accordingly. Supporting and auxiliary equipment requirements and deployment schedules are provided in the detailed equipment list.

Figure 16.95 Kamoa 5 truck requirement



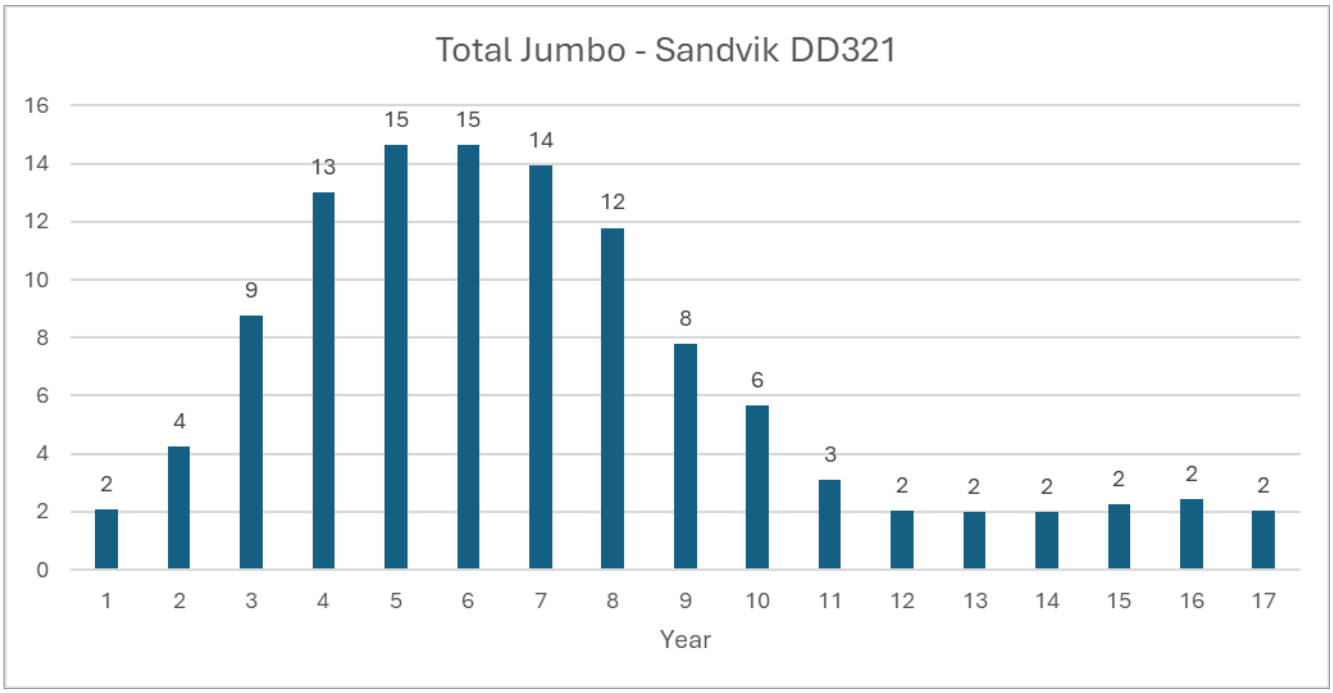
Source: AMC 2026

Figure 16.96 Kamoa 5 LHD requirement



Source: AMC 2026

Figure 16.97 Kamoa 5 development face drill requirement



Source: AMC 2026

16.11.5.6 Infrastructure requirements

Infrastructure requirements for Kamoa 5 are based on a conventional, decline-accessed underground mine. Twin declines provide primary access and truck haulage to surface, with ventilation supplied through surface intake and exhaust shafts and secondary ventilation established early via perimeter drives. Power, services, and dewatering infrastructure are progressively extended with lateral mine development as well as surface service droppers. No underground crushing, conveying, or paste backfill infrastructure is planned.

16.11.5.7 Materials handling

Material handling at Kamoa 5 is based on truck haulage via twin declines, with ore and waste hauled directly from underground working areas to surface. Truck loading is undertaken from perimeter drives and major access drives. Given the relatively modest tonnages, shallow mining depth, and large lateral mining footprint, no centralized hoisting shaft or conveyor decline has been planned.

16.11.5.8 Escapeways and emergency provisions

Escapeways at Kamoa 5 are designed for safety and redundancy, ensuring that all active working areas have interconnected routes via perimeter and access drives. This configuration provides multiple means of egress in the event of an emergency, allowing personnel to evacuate efficiently from any location within the mine.

To further enhance safety, refuge chambers are strategically positioned throughout the mine. As development progresses, these chambers are relocated as necessary to maintain effective coverage. The layout ensures that no active heading is more than 500 metres from a refuge chamber, providing workers with accessible temporary shelter should an emergency situation arise.

For emergency evacuation and response, a stench gas system is installed at the portal. This system supports rapid notification and direction during an incident, ensuring personnel are promptly alerted and can move safely towards designated escapeways and refuge chambers.

16.11.5.9 Ventilation and cooling requirement

The ventilation system at Kamoa 5 is designed to ensure a safe and healthy underground environment, tailored to the mine's operational scale and shallow depth. Primary ventilation is provided through dedicated surface intake and exhaust shafts, established in conjunction with the development of twin declines that serve as the main access and haulage routes. As the mine expands, the primary ventilation infrastructure is progressively extended to keep pace with lateral development.

Secondary ventilation is established via perimeter drives, ensuring that upcoming production has adequate airflow. This network of secondary ventilation enables effective distribution of fresh air and the removal of contaminants throughout the mine. The design allows for flexibility, so that as mining progresses, ventilation routes and controls can be adjusted or expanded as required.

At present, no dedicated cooling systems are planned for Kamoa 5, as the mine's relatively shallow depth and modest tonnage mean that natural ventilation and heat dissipation are sufficient to maintain a workable underground climate. However, this approach may be reviewed in the future should mining conditions change or production rates increase, necessitating the introduction of active cooling measures.

16.11.5.10 Water management

Groundwater and operational water are collected via drain holes reporting to a dedicated dewatering drive located below the orebody, from where water is pumped to surface for management or disposal.

Potable and process water required for mining operations is supplied from surface tanks and delivered to underground areas as necessary.

16.11.5.11 Power requirement

Power distribution at Kamoa 5 is tailored to suit the mine's large lateral footprint, with electrical supply fed both through the main portal and via surface droppers directly connected to the nearest substations. As the mine expands, the reticulation network will be extended in step with development.

16.11.5.12 Other

Compressed air and the communication network will be distributed using the same approach as power, extending in line with mine development to ensure coverage and supply as required.

16.11.6 Fixed infrastructure

Fixed underground infrastructure at Kamoa 5 is limited due to the relatively small scale of the deposit and the shallow nature of the mining operation. No permanent underground workshops or explosive magazines are included in the current design, as equipment maintenance and explosive handling are planned to be managed from surface facilities, and underground tramming distances are relatively short.

The mine design allows for the use of underground service bays to support minor maintenance activities and small equipment breakdowns. These service bays utilize redundant or inactive stockpile bays where available, minimizing the need for purpose-built underground infrastructure. The satellite service bays are intended for short-duration, light maintenance tasks only, with major repairs, services and component changes continuing to be undertaken at surface facilities.

16.11.7 Recommendations and next steps

After the adoption of a 4 m mining height (4 mH) scenario, the pillar dimensions remain at 6 m × 6 m, which were originally developed for a 5 m mining height (5 mH) configuration. As a result, the current layout is conservative for the reduced mining height and is likely to over-allocate pillar material within each production panel.

To improve alignment between design assumptions and operating parameters, it is recommended that a fully resolved mine design be completed for a true 4 mH configuration, including optimization of pillar dimensions, panel layouts, and sequencing. This would enable a more accurate assessment of dilution, waste movement, and head grade, while ensuring that geotechnical stability and operational efficiency are maintained without unnecessary sterilization of ore.

A further optimization opportunity exists through re-running the Mineable Shape Optimizer (MSO) using a mining height consistent with the finalized pillar and drive dimensions. While this may reduce the overall mineable inventory, it is expected to produce a smaller and more selective mining footprint, lowering capital development requirements and prioritizing recovery of the thicker, higher-grade portions of the orebody. Despite a potential reduction in total tonnage, this approach may improve overall project economics through reduced dilution and lower development intensity.

16.12 Kamoa 6

16.12.1 Summary of relevant information

The Kamoa 6 deposit features a copper orebody that is predominantly stratiform and exhibits a shallow to moderately dipping profile. The thickness of the orebody generally falls within the range of 4.5 to 5.5 metres, although there are localized sections where the thickness increases to between 5.5 and 6.5 metres. In addition, isolated zones have been identified where the orebody exceeds 10 metres in thickness.

The dip of the orebody is typically shallow, measuring between 2 and 15 degrees along the north–south axis. However, in certain areas, particularly in the east–west direction, the dip steepens locally up to 25 to 40 degrees. The overall mining depth for Kamoa 6 ranges from approximately 200 metres to 300 metres below the surface. This places the Kamoa 6 deposit at a greater depth compared to Kamoa 5, which has important implications for mine access, ventilation requirements, and the selection of appropriate mining methods.

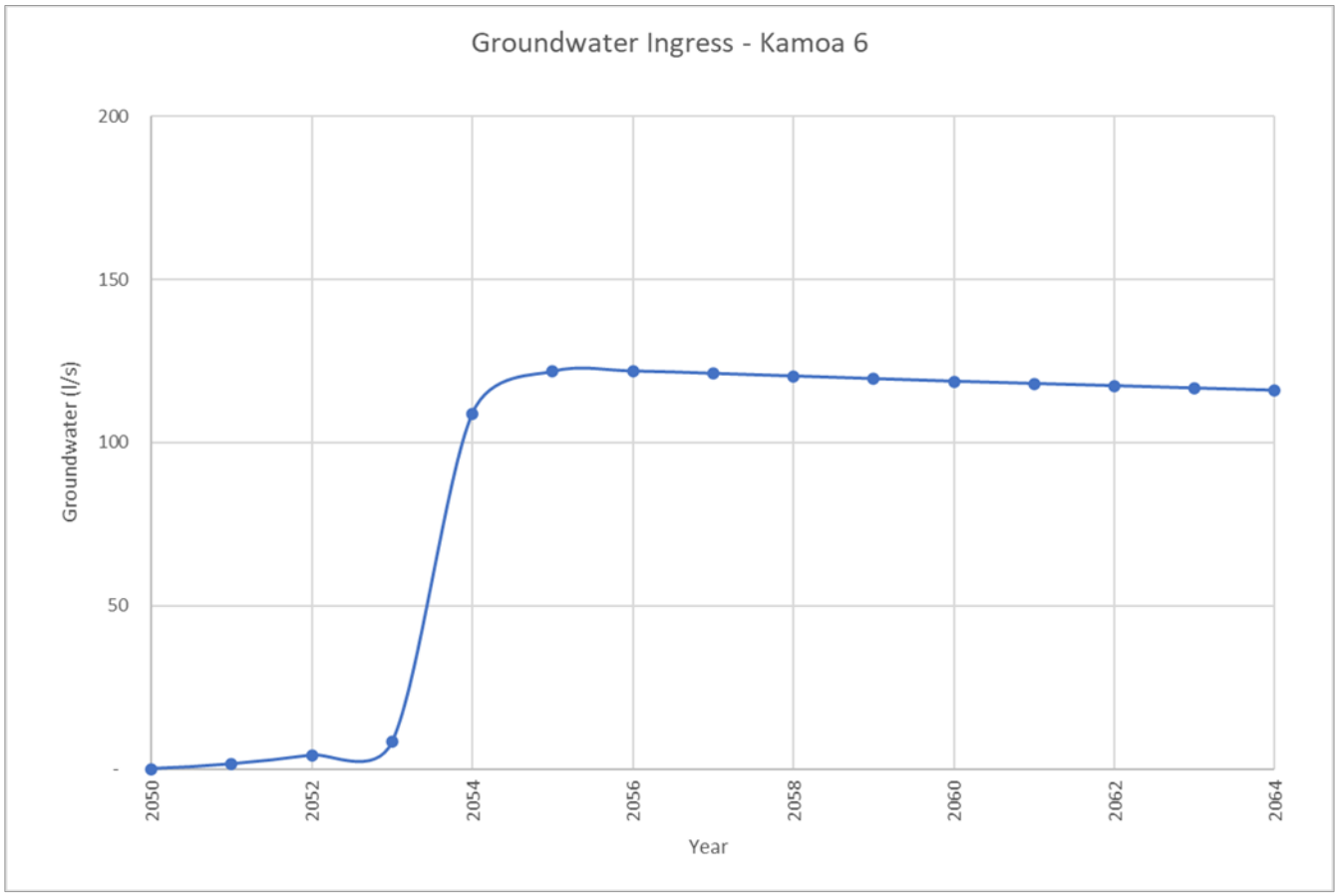
Mining method selection at Kamoa 6 is thickness-driven. Areas where the orebody exceeds approximately 6.5 m are planned to be mined using uphole primary and secondary stoping to improve recovery. Thinner zones are mined using Room-and-Pillar, adopting the same drive and pillar dimensions as Kamoa 5 to maintain consistency across operations. Stoping areas require paste backfill to enable secondary extraction, while R&P areas do not rely on backfilling.

16.12.2 Dewatering for Kamoa 6

Kamoa 6 will have a main decline pumping system located at the bottom of the decline. This system consists of a vertical dewatering dam that pumps water to the surface via dewatering boreholes. The main decline pumping arrangement is designed to handle 400 l/s.

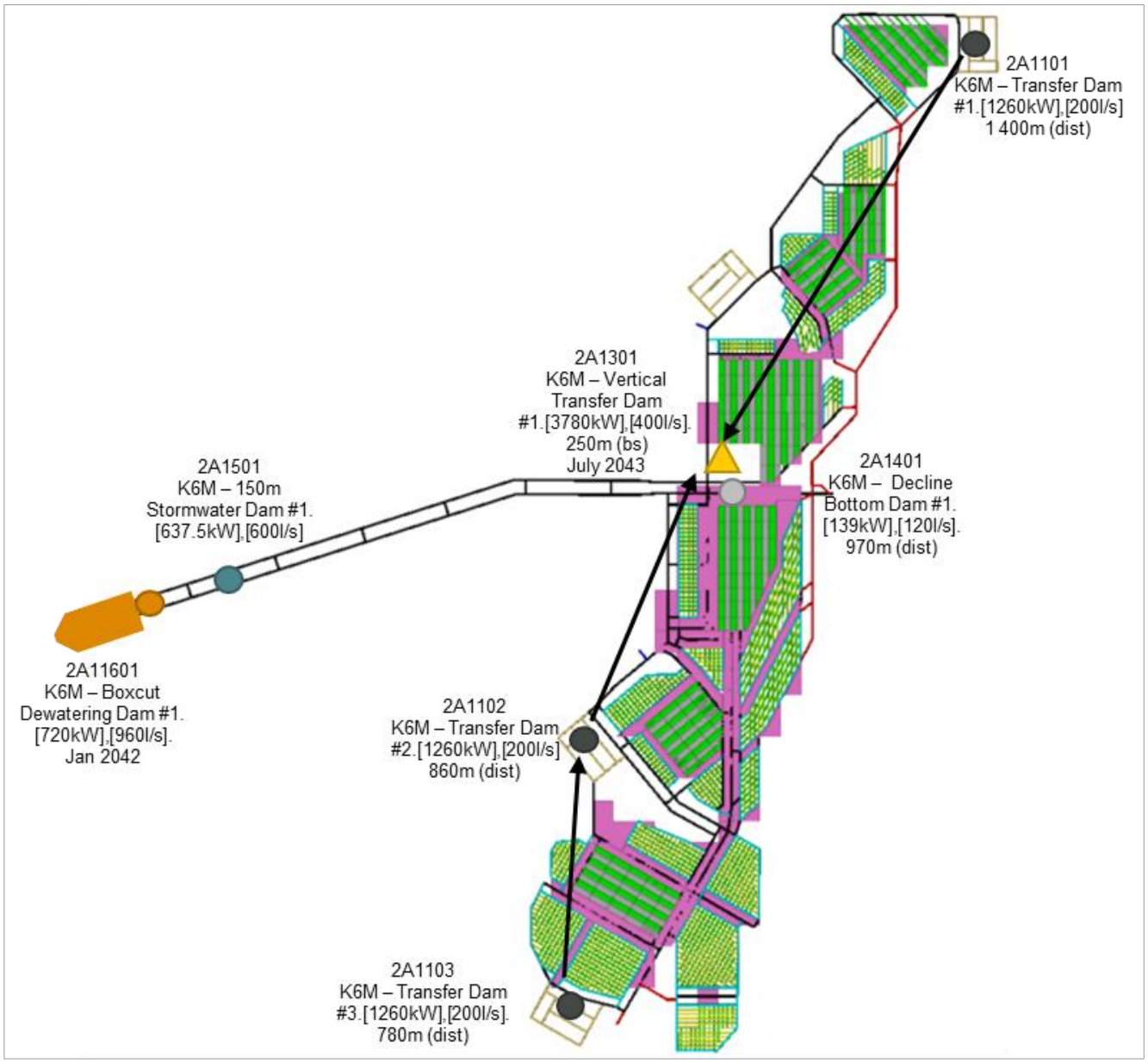
Kamoa 6 will have an expected peak water inflow rate of 120 l/s.

Figure 16.98. Kamoa 6 Groundwater Ingress



Source: WSP2026

Figure 16.99 Kamoa 6 Mine Dewatering Layout



Source: WSP2026

16.12.3 Kamoa 6 Ventilation

16.12.3.1 Kamoa 6 Ventilation strategy

The orebody will be accessed from the surface via the twin decline system as shown in Figure 16.100. Fresh air will be supplied to the mine from the decline and two 6.0mØ intake ventilation shafts located across the orebody. Guided by ventilation controls, the air will flow through the perimeter drives across the orebody to ventilate the development and stoping areas. Ore will be extracted by means of bord-and-pillar due to shallow depth of the orebody. The model shows the primary ventilation requirements of 623 kg/s at the peak production rate of 2.0 Mtpa, and a maximum depth of 228 m.

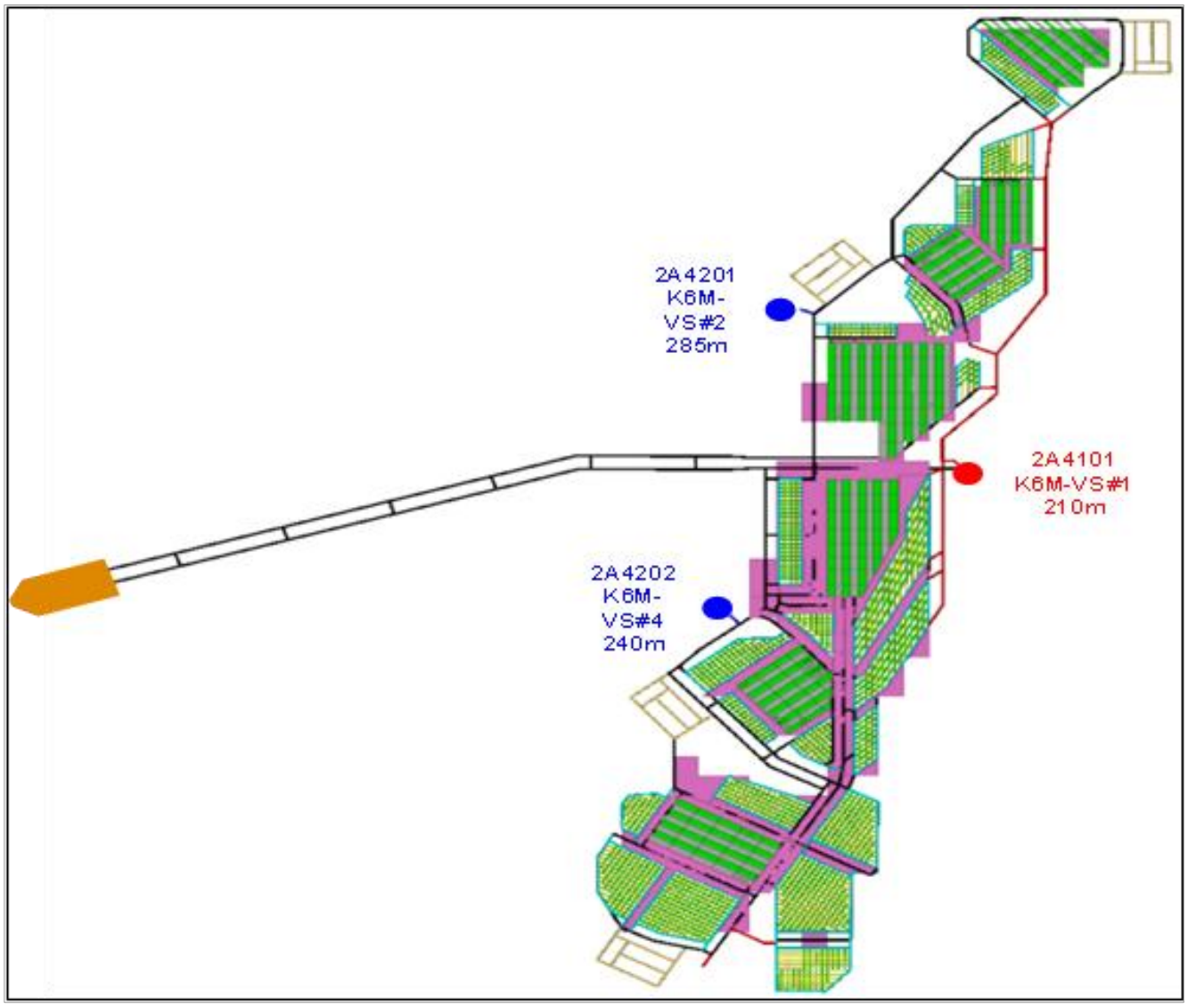
Figure 16.100 shows the mine layout with the location of ventilation upcast and downcast shafts. Air will return from south perimeter drives via three upcast shafts to surface. The model shows the primary ventilation requirements of 623 kg/s at the peak production rate of 2.0 Mtpa, and a maximum depth of 228 m.

Primary ventilation is provided through dedicated 2 surface intakes and 2 surface exhaust shafts, established in conjunction with the development of twin declines that serve as the main access and haulage routes. As mining progresses and production areas expand laterally, the primary ventilation infrastructure is progressively extended.

Secondary ventilation is established through access and perimeter drives, ensuring adequate airflow to active and planned production areas. This secondary ventilation network supports effective distribution of fresh air and removal of contaminants throughout both R&P and stoping areas.

At the current study stage, no dedicated cooling systems are included for Kamoa 6.

Figure 16.100 Kamoa 6 mine layout with ventilation shaft locations



Source DRA 2026

16.12.4 Kamoa 6 Production

The production plan for Kamoa 6 spans 9 years to produce 5.6 million tonnes of ore and 0.1 million tonnes of copper. The production profile is described below in Table 16.56

Table 16.56 Kamoa 6 production profile

Future operations	Ore tonnes (Mt)	Grade (%Cu)	Cu metal (Mt)	Peak production (Mtpa)	Steady production (Mtpa)	Life of production (Yrs)
Kamoa 6	7.2	2.7	0.1	1.3	0.6	9

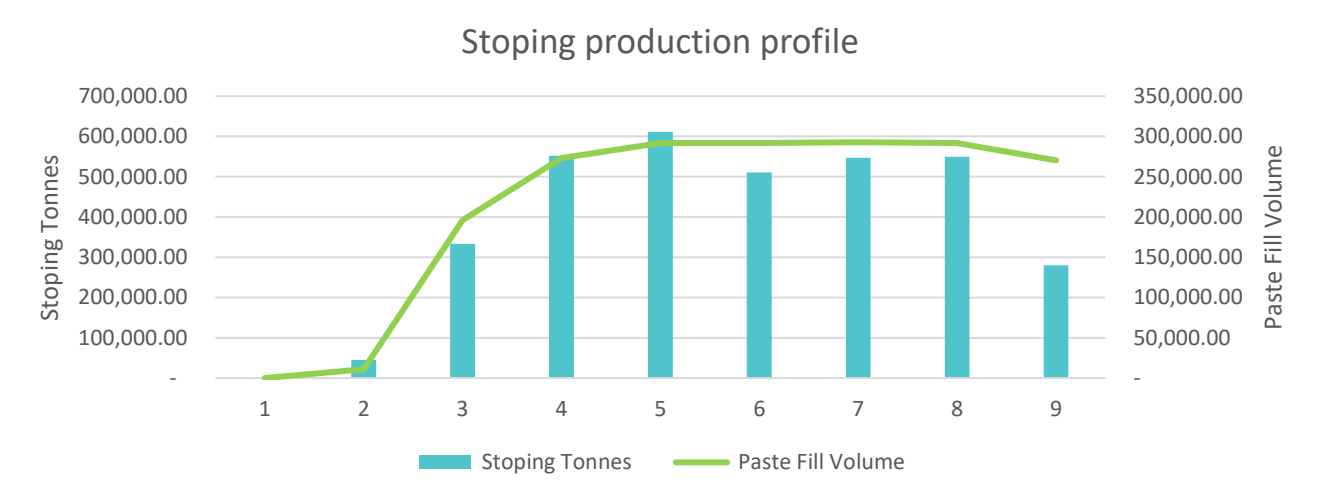
Equipment and task rates validated with the AMC SmartData™ benchmarking database applied to the production schedule are shown in Table 16.57.

Table 16.57 Kamoa 6 equipment and task rate

Equipment or task	Rate
Jumbo	130 m/month
Production drill	150 m/day
Raisebore	2 m/day
Loader - stopes	1200 t/day
Loader – development and pillar extraction	800 t/day
Cable bolter	150 m/day
Decline and major access	90 m/month
Lateral development (Other)	50 m/month
Paste backfill rate	800 m^3/day

The Kamoa 6 production profile reflects a planned strategy to prioritize higher-grade zones and stoping areas as soon as possible. Production ramps up rapidly from Year 2 to a peak of approximately 1.24 Mtpa in Year 5, driven by early access to stoping blocks that deliver both higher tonnes and higher head grades. As mining progresses and stoping areas are depleted, production transitions toward lower-rate R&P extraction from remaining areas, resulting in a gradual decline in both tonnage and grade toward the end of mine life. This sequencing is intended to front-load metal production and improve the project by bring positive cashflows in the early years.

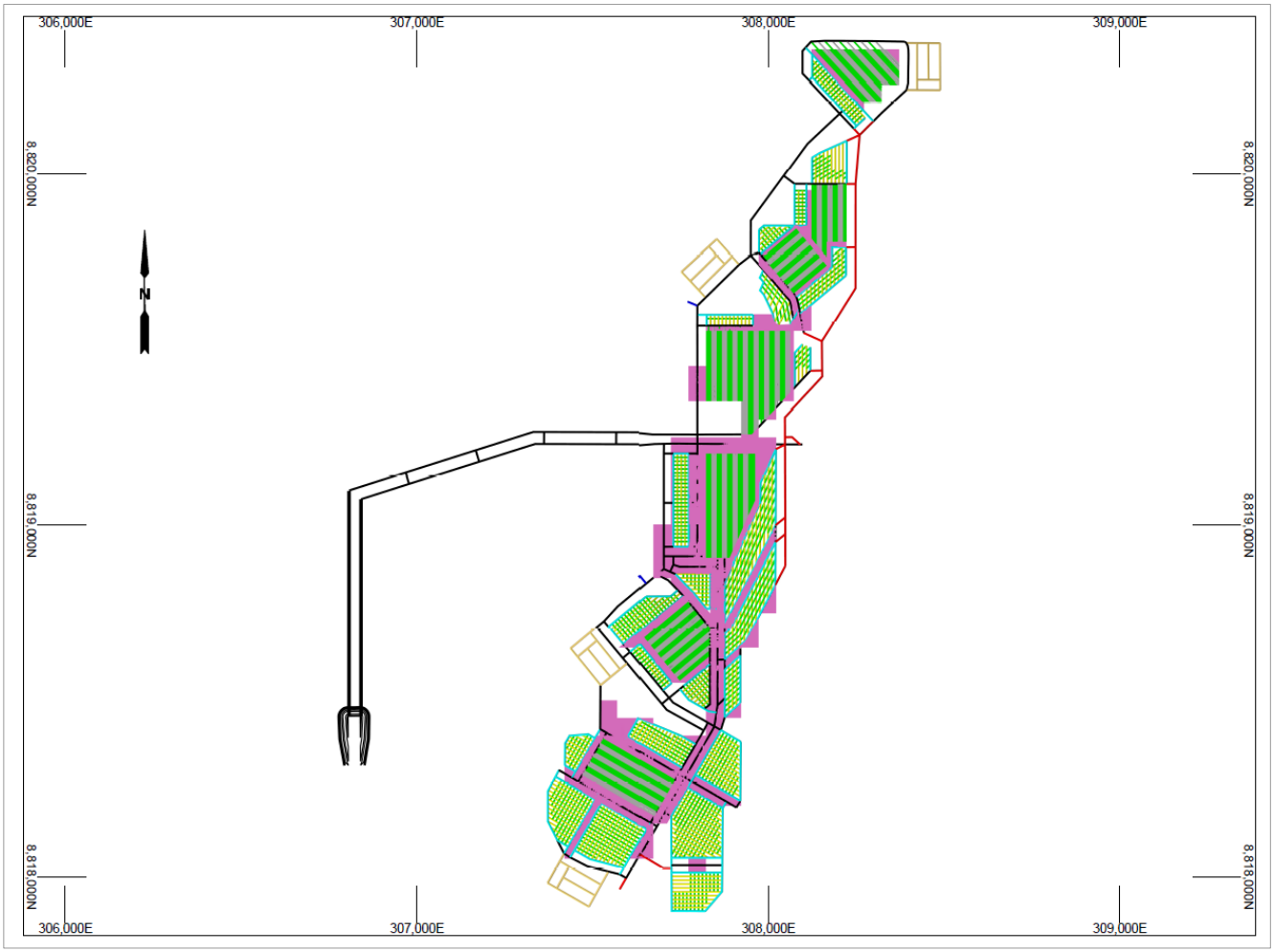
Figure 16.101 Kamoa 6 stoping profile



Source: AMC2026

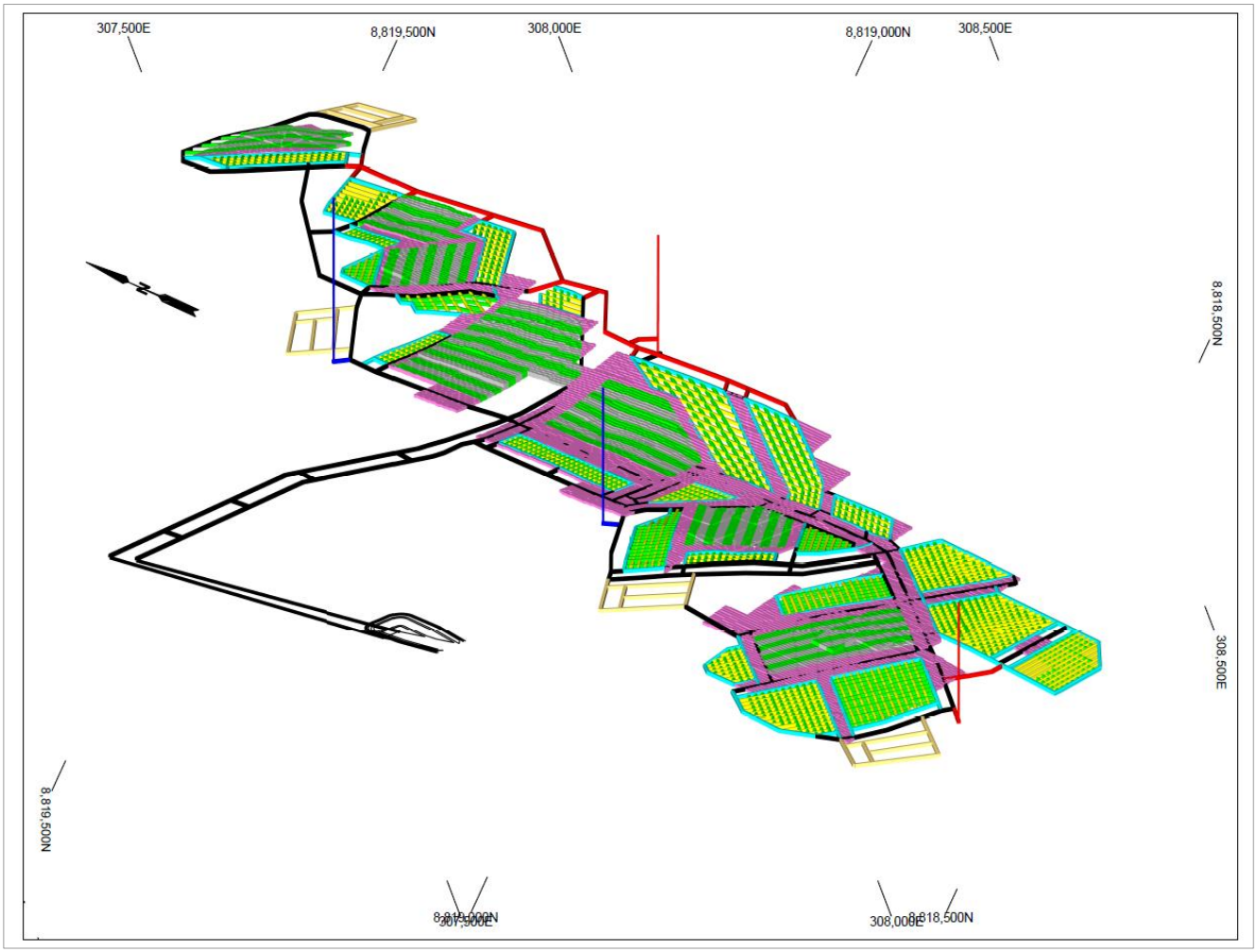
Stoping production ramps up rapidly to a peak during the mid-mine life, with up to 75–80 stopes mined per year and average stope tonnages of approximately 7–12 kt per stope. This results in a corresponding increase in paste backfill demand, which rises from minimal volumes in the early years to approximately 270,000–293,000 m^3 per year during peak stoping activity. As the number of stopes declines toward the end of mine life, both stoping output and paste backfill volumes reduce accordingly, reflecting the staged ramp-up and ramp-down of stoping operations.

Figure 16.102 Kamoa 6 plan view



Source: AMC2026

Figure 16.103 Kamoa 6 isometric view



Source: AMC2026

16.12.4.1 Production summary

Production at Kamoa 6 utilizes a hybrid mining method, employing R&P techniques in thinner ore zones and uphole primary and secondary stoping in areas where the orebody thickness exceeds approximately 6.5 meters. Stope design parameters are standardized to typical geometries of roughly 15 meters in width by 40 meters in length and 7–15 meters in height. Mining operations proceed according to a structured sequence: primary stopes are extracted first, while secondary stopes are mined only after the adjacent primaries have been backfilled and adequately cured. In R&P zones, extraction follows a perimeter development, panel access, and retreating crosscut sequence consistent with sequence implemented at Kamoa 5.

16.12.4.2 Modifying factors

Table 16.58 Kamoa 6 modifying factors

Activity	Dilution factor	Mining recovery factor
Horizontal development	1.1	0.95
Vertical development	1.0	1.0
Primary stope	1.15	0.92
Secondary stope	1.2	0.9

Table 16.59 Other production assumptions

Other assumptions	Value	Comment
Stope drill Metre/t	8 t/m	Production drilling metre calculation
Stoping longhole and cable re-drill factor	1.1	Applied to total production and cable drilling metres
Stope brow cable	36 m	6 x 6 m cables
Intersection cable	36 m	6 x 6 m cables
Paste filling mining distance	10 m/stope	Paste wall / bulkheads are built 10 m back from stope brow
Stope CMS – Paste wall construction	2 days	2 days delay assumed for stope CMS and paste wall construction before filling commences
Paste cure	10 days	10 days paste curing time before paste mining can start
Paste fill holes drilling	80 m/stope	Two holes per stope, Fill hole is 3 passes, including reaming to required, 3 x 20 m. Breather hole is 20 m, single pass.

16.12.4.3 Kamoa 6 development and infrastructure

Primary access is provided via twin declines, with lateral development extending to establish production domains for both R&P and stoping areas. Infrastructure is staged in line with mine development and production ramp-up, including the installation of ventilation shafts, services, and localized dewatering infrastructure to support increasing ventilation and water management demand.

Paste backfill infrastructure at Kamoa 6 comprises a surface paste plant located near the centre of gravity of the orebody, reflecting the distribution of stoping areas along the north–south strike. This location reduces reticulation distances and supports efficient delivery to multiple production areas. Due to the relatively shallow mining depth, gravity flow alone is insufficient to provide the required paste delivery pressure. As a result, surface displacement pumps are required, together with multiple surface paste dropper boreholes to deliver paste to the underground stoping areas.

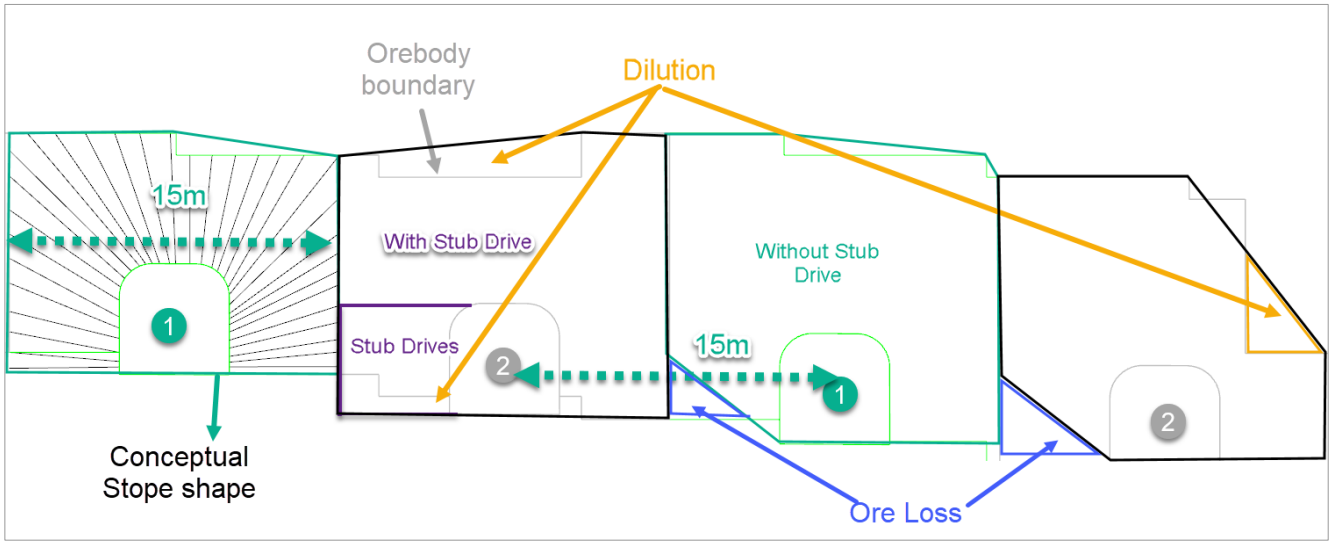
16.12.4.4 Basis of Design

Production at Kamoa 6 utilizes a hybrid mining method, employing R&P techniques in thinner ore zones and uphole primary and secondary stoping in areas where the orebody thickness exceeds approximately 6.5 meters. Stope design parameters are standardized to typical geometries of roughly 15 meters in width by 40 meters in length and 7–15 meters in height. Mining operations proceed according to a structured sequence: primary stopes are extracted first, while secondary stopes are mined only after the adjacent primaries have been backfilled and adequately cured. In R&P zones, extraction follows a perimeter development, panel access, and retreating crosscut sequence consistent with sequence implemented at Kamoa 5.

All stopes at Kamoa 6 are designated as blind uphole stopes (Figure 16.104). To optimize ore recovery, sub-drilling is used at the stope backs. However, this can cause overbreak at the stope backs, which increases dilution.

At the drilling and bogging horizon, ore is assumed to rill downward at an average angle of approximately 45°. Where stub drives or hammer heads are not developed, ore located outside this rill angle may remain in the stope sidewalls, resulting in ore loss. The development of hammer heads or stub drives can improve recovery by capturing this material; however, in some cases hammer head development may extend into waste rather than ore, introducing additional planned dilution. As a result, the decision to implement hammer heads or stub drives is treated as a stope-specific design consideration, with drill-and-blast engineers required to evaluate the trade-off between maximizing recovery or avoiding unnecessary dilution on a case-by-case basis.

Figure 16.104 Stope cross section view



Source: AMC2026

Dilution estimate applied for the Kamoa 6 stope design:

- 15% for primary stopes
- 20% for secondary stopes (due to firing against paste-filled voids)

Corresponding recovery factors are set at:

- 92% for primary stopes
- 90% for secondary stopes

These adjustments reflect anticipated ore loss under current operational parameters.

Table 16.60 Excavation profile

Description	Profile	Width (m)	Height (m)	Gradient / Dip
Major access	Arched	6	6	Variable
Sump	Arched	6	6	1:7 DOWN
Connection drive	Arched	6	6	Variable
Decline	Arched	6	6	1:7 UP / DOWN
Fresh air way	Arched	6	6	1:50 UP
Ore drive	Square	5	5	Variable
Panel entry drive	Arched	5	5	Variable
Perimeter drive	Arched	6	6	Variable
Return air way	Arched	6	6	1:50 UP
UG return air raise	Square	2.5	2.5	90°
Surface ventilation shaft	Circle	3.5Ø		90°
Crosscut	Arched	5	5	Variable

16.12.5 Development mining and sequence profile

Total development accelerates from Year 2, peaking between Years 5 and 6 at approximately 14.1–14.3 km annually. This increase is predominantly attributable to operating horizontal development, which reaches 12.8–13.5 km per year during the peak period. The high rate of operating horizontal development results directly from the R&P mining method, which is development-intensive across multiple production zones.

Initial capital horizontal development is concentrated in Years 1 through 4, with a significant decrease after Year 5 as major access routes and essential infrastructure are completed. This trend demonstrates a transition from capital expenditure to ongoing operational expenditure for sustaining production. Capital vertical development remains limited, amounting to approximately 731 meters constructed in Years 2 to 4 for surface ventilation shafts, representing only about 1% of total development activity.

16.12.5.1 Infrastructure requirements

Infrastructure requirements for Kamoa 6 are based on a conventional, decline-accessed underground operation supporting a hybrid mining approach comprising R&P and backfill-supported stoping. Primary access and material handling are provided via twin declines, with truck haulage of ore and waste directly to surface. The mine is subdivided into multiple ventilation districts, supported by surface fresh-air intake shafts and exhaust shafts installed progressively as mining advances. Dewatering is managed through localized pump stations within production domains

Backfill infrastructure is required for stoping areas and comprises a surface paste plant centrally located relative to the orebody, together with surface displacement pumps and multiple paste dropper boreholes to distribute paste underground. Power, water, and other services are reticulated along main access drives and extended progressively with mine development. Given the laterally extensive nature of the orebody, additional power, water, and service installations may also be supplied directly from surface via dedicated service boreholes where required. No underground crushing, conveying, or hoisting infrastructure is included in the current design

Materials handling

Material handling at Kamoa 6 is based on truck haulage via twin declines, with ore and waste loaded by LHDs from active stoping and R&P areas and hauled along primary access drives directly to surface. Haulage routes are designed to provide efficient access to multiple production areas, with short internal tramming distances supported by the laterally extensive mine layout. Surface handling facilities are planned to receive and manage run-of-mine material, with no underground stockpiles or ore bunkers included in the current design.

At the current study stage, no underground conveyors or hoisting shafts are planned, given the relatively short mine life and moderate production tonnages. The layout retains flexibility to accommodate future material handling upgrades, including conveyors or shafts, should mine life extensions or increased production rates justify such infrastructure in future studies.

Escapeways

Escapeways at Kamoa 6 are designed for safety and redundancy, ensuring that all active working areas have interconnected routes via perimeter and access drives. This configuration provides multiple means of egress in the event of an emergency, allowing personnel to evacuate efficiently from any location within the mine.

To further enhance safety, refuge chambers are strategically positioned throughout the mine. As development progresses, these chambers are relocated as necessary to maintain effective coverage. The layout ensures that no active heading is more than 500 metres from a refuge chamber, providing workers with accessible temporary shelter should an emergency situation arise.

For emergency evacuation and response, a stench gas system is installed at the portal. This system supports rapid notification and direction during an incident, ensuring personnel are promptly alerted and can move safely towards designated escapeways and refuge chambers.

Water management

The inclusion of stoping operations introduces additional sources of water inflow compared to R&P only mining. To manage this, localized sumps are planned within each production domain to collect water generated from the mining activities.

The operational strategy for these sumps, including whether water is temporarily stored underground or pumped directly to surface facilities, will be assessed in the next study phase, following further evaluation of inflow rates, pumping requirements, and surface water management capacity.

Power requirement

Power distribution at Kamoa 6 is designed to accommodate the laterally extensive nature of the orebody and distributed production areas. Electrical supply is delivered via the main portal and, where required, supplemented by surface service droppers connected to nearby substations. As mining expands, the power reticulation network is extended progressively in line with development.

Other

Compressed air and communication services are distributed using the same approach as power, with infrastructure extended progressively as mining advances to ensure adequate coverage and reliability across all active mining areas.

Fixed infrastructure

Fixed underground infrastructure at Kamoa 6 is limited and reflects the moderate scale of the operation and the use of truck-based material handling. No permanent underground workshops or explosive magazines are included in the current design, with equipment maintenance and explosive handling planned to be managed from surface facilities. This approach minimizes underground capital requirements while maintaining operational flexibility.

The mine design allows for the use of underground service bays to support minor maintenance activities and small equipment breakdowns. These service bays utilize redundant or inactive stockpile bays where available, minimizing the need for purpose-built underground infrastructure. The satellite service bays are intended for short-duration, light maintenance tasks only, with major repairs, services and component changes continuing to be undertaken at surface facilities.

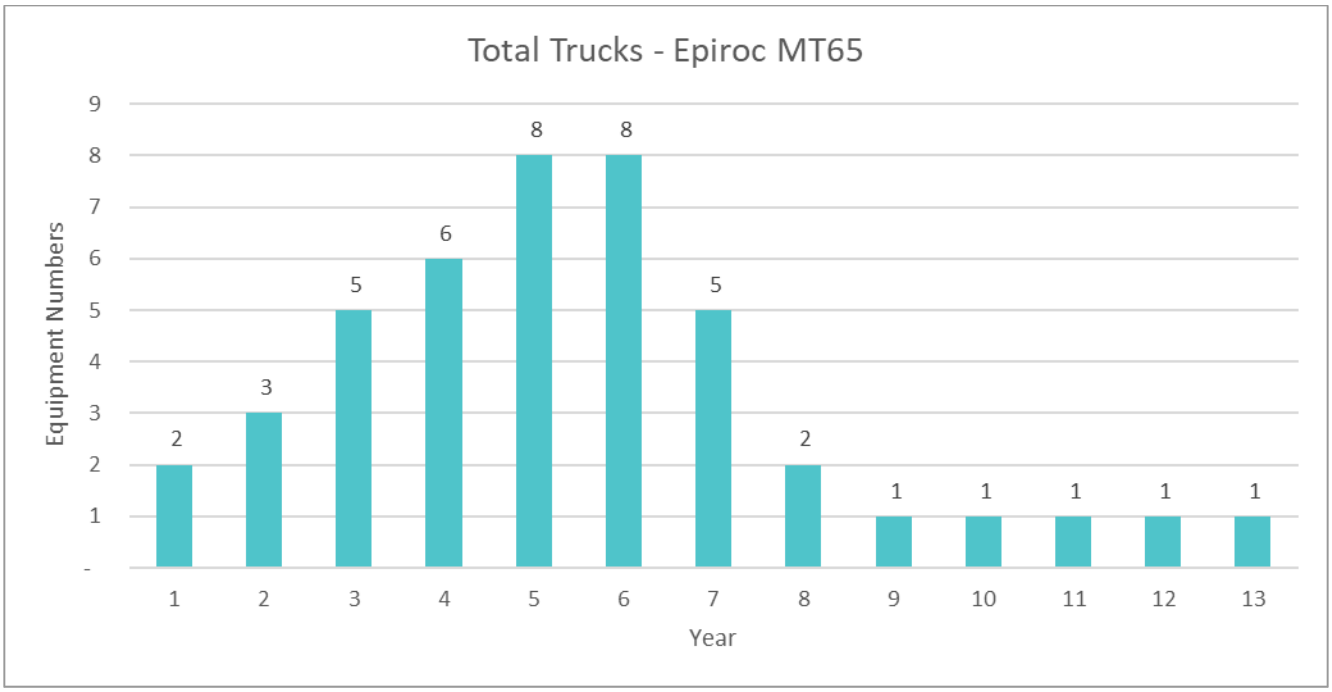
16.12.6 Kamoa 6 equipment requirements

16.12.6.1 Mobile equipment fleet

Please refer to Figure 16.105, Figure 16.106 and Figure 16.107 for planned equipment requirements throughout the mine life.

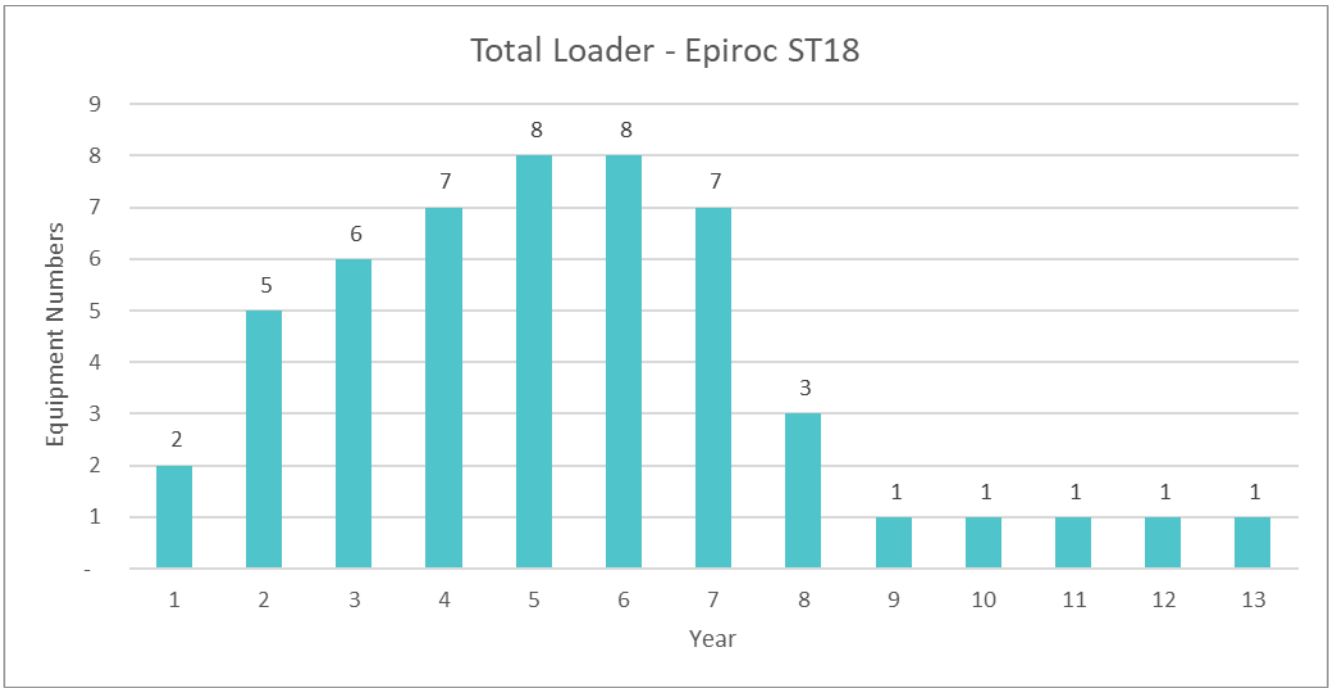
Truck and LHD requirements increase steadily during the early years as stoping and R&P production areas are established, peaking during the mid-mine life when production and development activity are at their highest. Drilling requirements are dominated by development face drilling, reflecting the development-intensive nature of the R&P mining method, with a smaller number of production longhole drills deployed to support stoping and pillar extraction. As mining activity reduces toward the end of mine life, fleet numbers decrease accordingly. Supporting and auxiliary equipment requirements and deployment schedules are provided in the detailed equipment list.

Figure 16.105 Kamoa 6 truck requirement



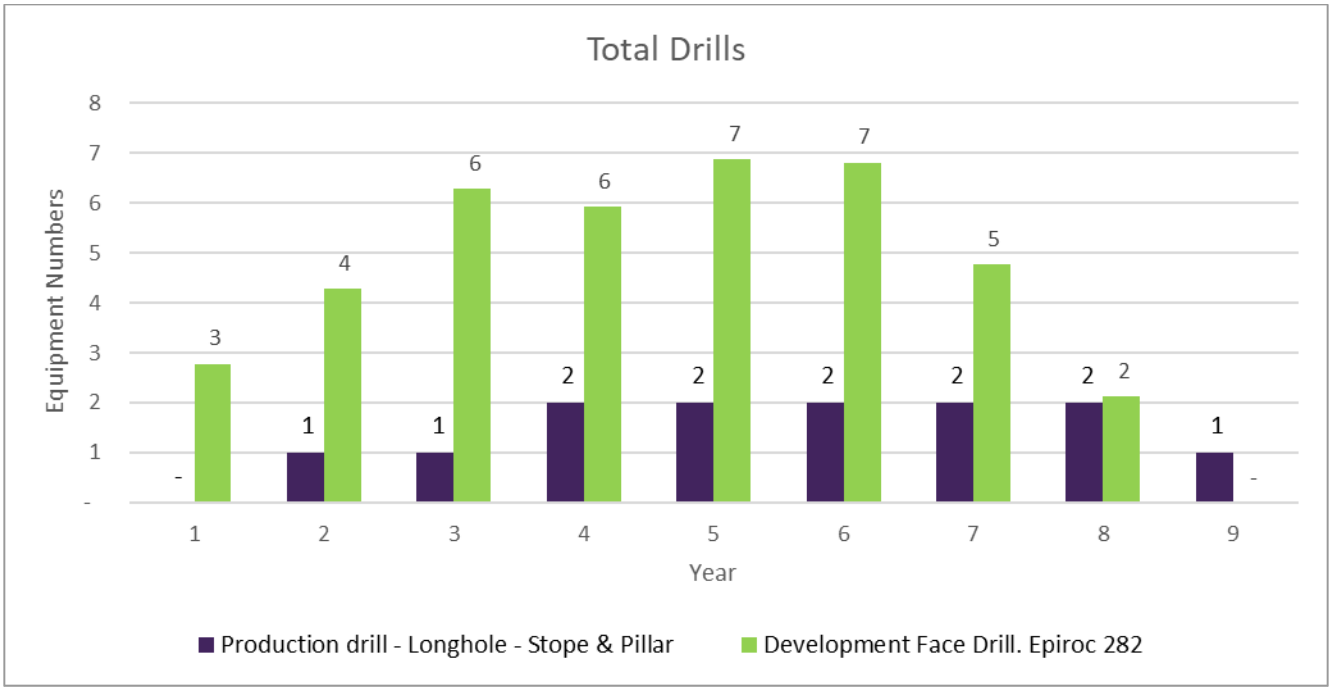
Source: AMC 2026

Figure 16.106 Kamoa 6 LHD requirement



Source: AMC 2026

Figure 16.107 Drilling units



Source: AMC 2026

16.12.6.2 Fixed equipment

Fixed equipment at Kamoa 6 is staged in line with mine development and production ramp-up. The primary fixed installations relate to the ventilation system, with surface intake and exhaust shafts developed progressively to support expanding mining areas and increased ventilation demand. Associated fixed infrastructure, including fans and permanent ventilation controls, is installed as each shaft becomes operational.

Additional fixed infrastructure includes paste backfill distribution systems required to support stoping operations, comprising surface paste plant, paste droppers, underground steel paste reticulation, dumping facilities, and manual or automated divert valves at each surface dropper location to control paste delivery to individual stopes.

No fixed underground crushing, conveying, or hoisting equipment is included in the current design.

16.13 Recommendations and next steps

The orebody thickness is shown in Figure 16.108 and identifies distinct thickness domains, with sharp boundaries between thinner and thicker zones without transition areas. This pattern is unlikely to fully reflect natural geological variability and suggests that the current thickness model is strongly influenced by drill spacing and interpolation, rather than fully constrained by data.

Additional infill drilling is therefore recommended to better define orebody thickness continuity and variability along strike and down dip. Improved definition of thickness distribution will reduce uncertainty and directly support appropriate refinement of mining method selection and associated dilution and recovery assumptions.

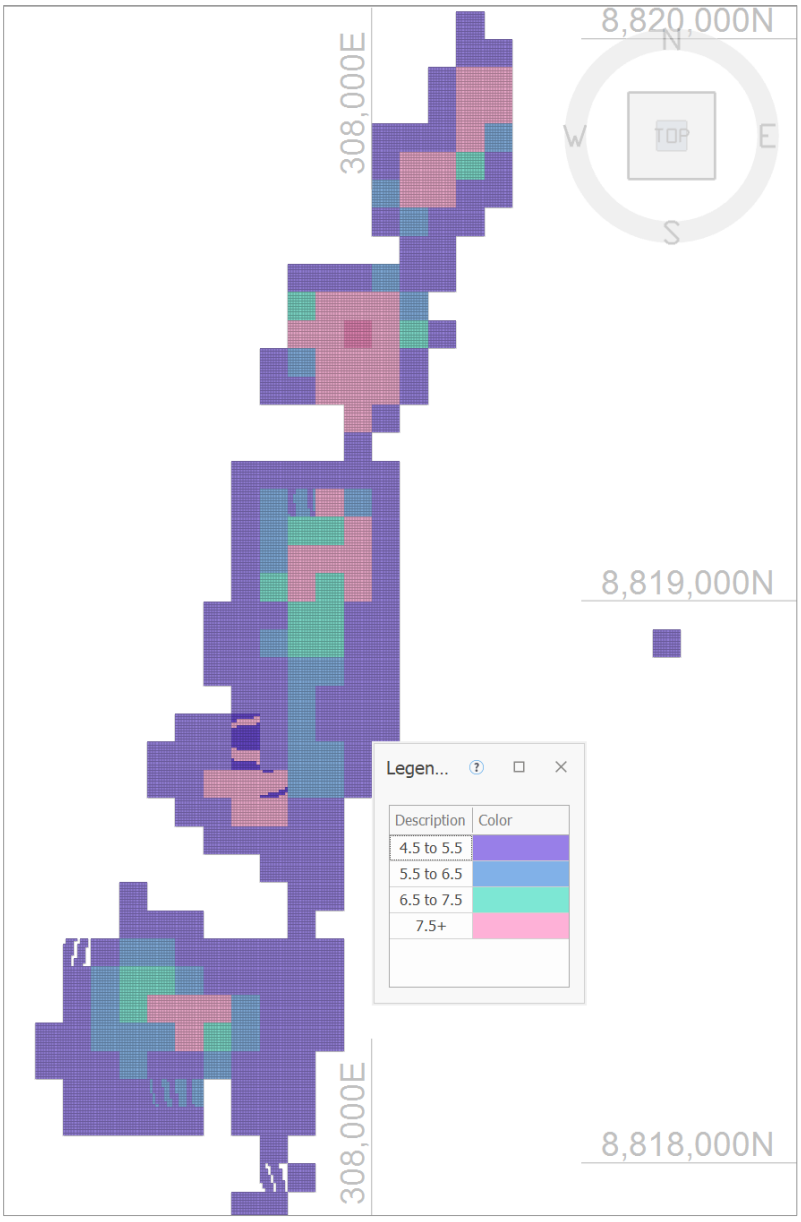
If further drilling demonstrates that the orebody is consistently thicker and laterally continuous, a greater proportion of the deposit may be suitable for conventional stoping with top access and tight backfilling, which would improve control of recovery, dilution and operational safety. Under this scenario, mine access could be preferentially located on the eastern (footwall) side of the orebody to improve infrastructure stability, and the case for a surface paste plant would be strengthened by increased reliance on backfill-supported stoping.

Based on the current mine plan, stoping productivity at Kamoa 6 is constrained by paste backfill capacity, with the paste plant throughput currently capped at approximately 800 m^3 per day. This limitation represents a potential bottleneck during periods of peak stoping activity, when annual backfill demand approaches 270,000–293,000 m^3. Further optimization of the production profile or evaluation of increased paste plant capacity is therefore recommended to ensure backfill availability does not constrain stoping rates during the main production period.

Conversely, if additional drilling confirms that the orebody remains predominantly thin or highly variable, a larger proportion of the deposit would be more suitably mined using R&P, potentially reducing the reliance on expensive paste infrastructure. In this case, the requirement for a paste plant could be reduced or eliminated, simplifying infrastructure and lowering capital expenditure.

Overall, the next phase of drilling and geological refinement is critical to define the orebody thickness to a higher resolution, reduce design uncertainty, and enable a more confident and optimized selection of mining method and supporting infrastructure.

Figure 16.108 Kamoa 6 orebody thickness distribution



Source: AMC 2026

16.14 Kakula West - East

16.14.1 Summary of relevant information

The Kakula West deposit has been separated into two sections broadly defined by the boundary of operationally achievable mining utilizing the Drift and Fill (D&F) mining method. They are now named as Kakula West – East (KW-E) and Kaula West – West (KW-W). Gradients for D&F are to be constrained to a maximum of <15%. This separation exists at a structural transition zone in the middle of the orebody, where it moves from a relatively consistent dip on the east to more complex and steeper geometry on the west. For simplicity, current mine plans consider these mining fronts as largely separate entities with some common links for sharing infrastructure. Future mine plans should look to leverage the two assets and combine them wherever possible.

KW-E has a depth of between approximately 650 m and 1,000 m below surface. The orebody thickness is up to a maximum of 12 m. D&F has been re-selected, in line with previous studies, to utilize stope production within the orebody thereby reducing development intensity, introduce paste fill cycles that minimize cycles and create separate, manageable working fronts.

The dip of the orebody ranges from flat to approximately 40° from horizontal. Gradients within the orebody are managed by traversing in the dip direction. The included angle for traversing the orebody is consistent with previous studies at 25° between drives, resulting in acute angle bull nose pillars. It is anticipated that specific breakaway sequences and increased support will be required to ensure stability. Drive sizes are intended to enable access but minimize development workload with the resulting intersection size requiring additional support through the use of cable bolts.

A major change from the prior study is the inclusion of a designated access and haul route on the extents of the mineralization. It provides a uniform gradient access to the orebody for material movement via a dedicated twin drives. This in turn this minimizes the internal orebody traffic and the number of drives required, maximizing ore recovery.

Water inflows into the mine are likely to be substantial, exceeding practical limits for staged submersible pump systems. A drainage horizon has been incorporated into the design to enable dewatering through boreholes. Pumping infrastructure will be integrated into the drainage level to support dewatering. That additional benefit of this horizon is that it provides additional flooding protection should prolonged power loss occur. Design of infrastructure should aim to maximize this capacity.

Kakula West lies to the west of the West Skarp Fault structure set, which consists of three sub parallel faults. A fourth likes approximately 1,100 m to the west of the West Skarp Fault, cutting the KW-E D&F region. This area is also one of the deepest mining areas within the Kamoa / Kakula mining area. Improving geological and geotechnical knowledge of these fault complexes for future studies will improve confidence in the mine plans in these regions. Currently, a 20 m pillar has been left on either side of the fault to account for anticipated poor ground and challenging recovery.

Mine access is currently planned via portal.

Initial orebody access is via decline to the top of the KW-E orebody at 650 m depth. This access will be required for the life of the operation. The primary access drives to the north and south of the orebody are offset from production by an 80m pillar. This is to ensure life-of-mine stability of the primary access drives. This should be evaluated in future studies to confirm and optimize the standoff.

The geotechnical design for Kakula West – East incorporates several key elements to optimize stability and ore recovery. Twinned drives are utilized to maintain robust pillars between excavations, providing structural support and ensuring safe access throughout the mine. These drives are strategically offset to the drainage level, which facilitates effective dewatering and helps manage substantial water inflows by integrating pumping infrastructure and drainage horizons, minimizing flooding risk and improving buffer before flooding production areas.

Intersection cables are planned for within the design as a standard for all intersections. This is especially necessary within the KW-E case due to the 25° internal angle of turnouts whilst on the orebody. Bull nose pillars are a regular feature and it is anticipated that specific intersection sequencing will be employed to manage overbreak.

Primary vertical rises are offset from accessways by a minimum of 20 m. All rises are to be combined with brow cables for support. Rises are anticipated to be unsupported. Limited internal rises within the footprint have been factored. Positioning of these rises is to support development advance of the drainage horizon and initial northward development on the orebody until the primary return connections are established.

Geotechnical guidance has been adopted for the mining sequence and layout of the KW-E orebody and its application of drift and fill. This guidance has been adapted to the acute angle layout applied to KW-E ensuring the local pillar dimensions are maintained. It is anticipated that changes to this layout will occur in future studies as more bespoke design is possible.

Stope cable bolts are installed to reinforce the orebody hangingwall, especially in areas where development sizes are minimized to reduce excavation intensity. Backfill is introduced into mined-out stopes, not only to support the back but also to create separate, manageable working fronts and minimize mining cycles. Chain and inter-panel pillars are incorporated into the block layout, serving as additional barriers against ground movement and enhancing the integrity of the overall mine structure. Together, these measures reflect a comprehensive geotechnical approach designed to address the unique challenges presented by depth, fault zones, and water management at Kakula West – East.

16.14.2 Hydrogeological Setting for Kakula West - East

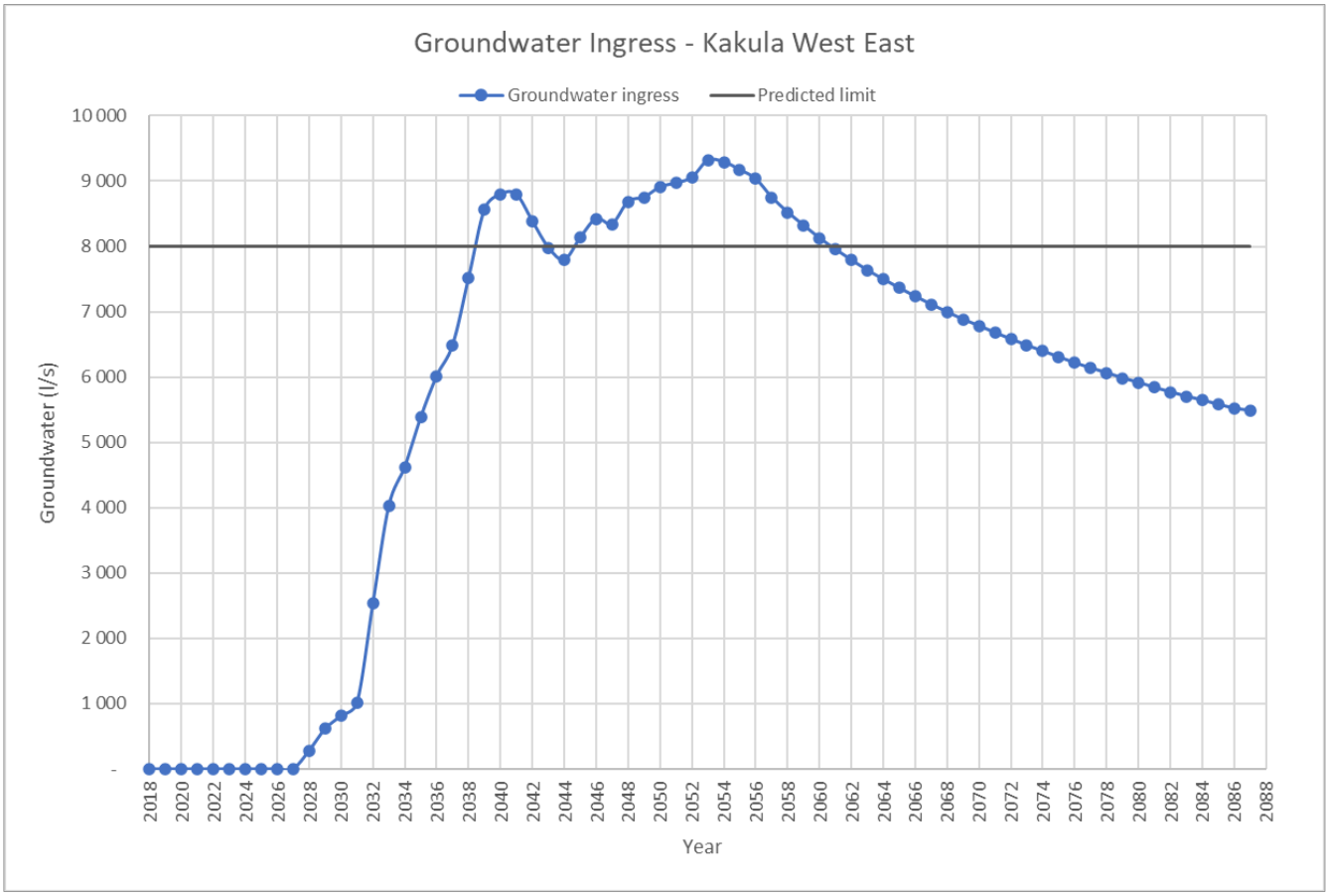
The West Skarp Fault and its sympathetic sub-parallel structures intersect the major aquifers local to the Kakula West – East deposit. The KW-E mine is bound by the West Skarp Fault and modelling indicated a maximum mine water inflow of approximately 4,500 L/s. The hydrogeological setting is outlined in Section 16.2.6 with specific reference to Figure 16.43.

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 16.110. 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.

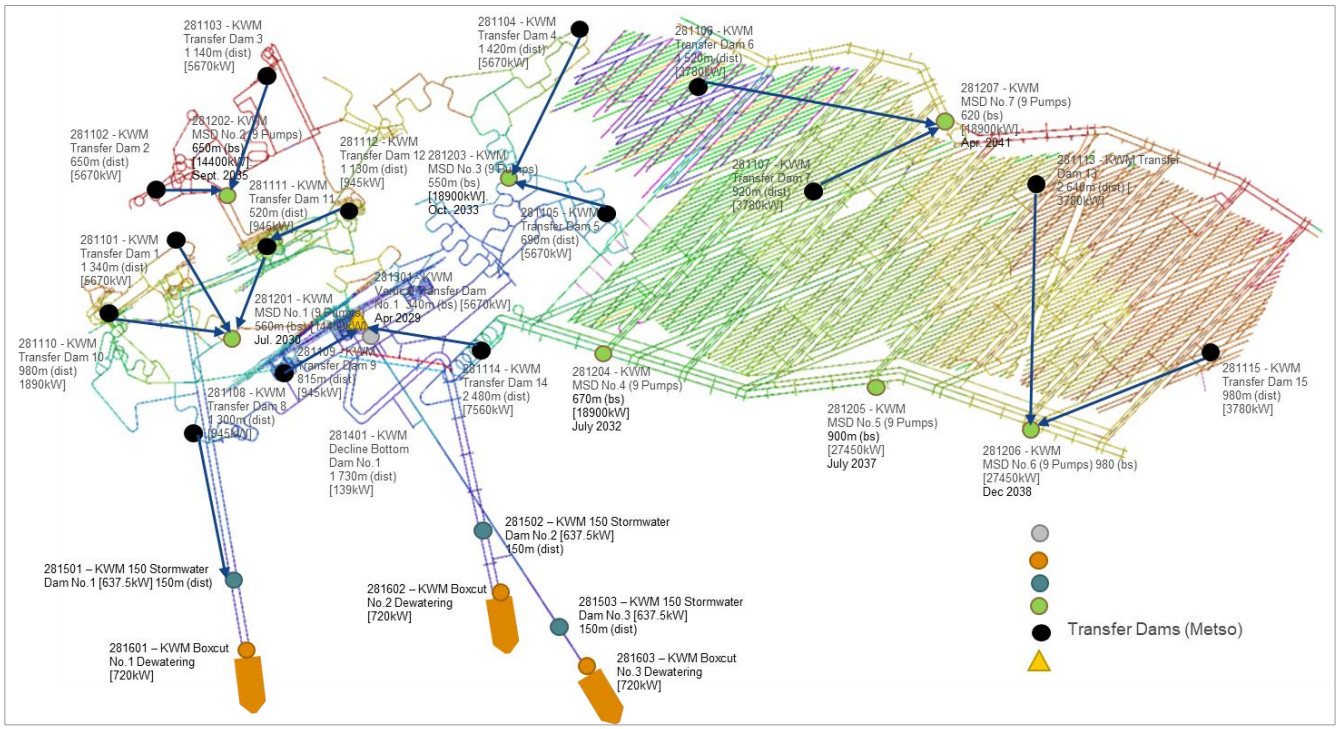
Each multistage dewatering pump station will be fed by two or three transfer dam pump stations. The transfer dam pump stations, indicated as black dots in Figure 16.110, will have the capacity to pump 200 – 1,000 l/s to the main multistage dewatering dams. The transfer dams will be strategically positioned to cover the required footprint of the mined ore body.

Figure 16.109 Kakula West (East Section) groundwater ingress



Source: WSP

Figure 16.110 Kakula west mine dewatering layout



Source: WSP

16.14.3 Kakula West East Ventilation

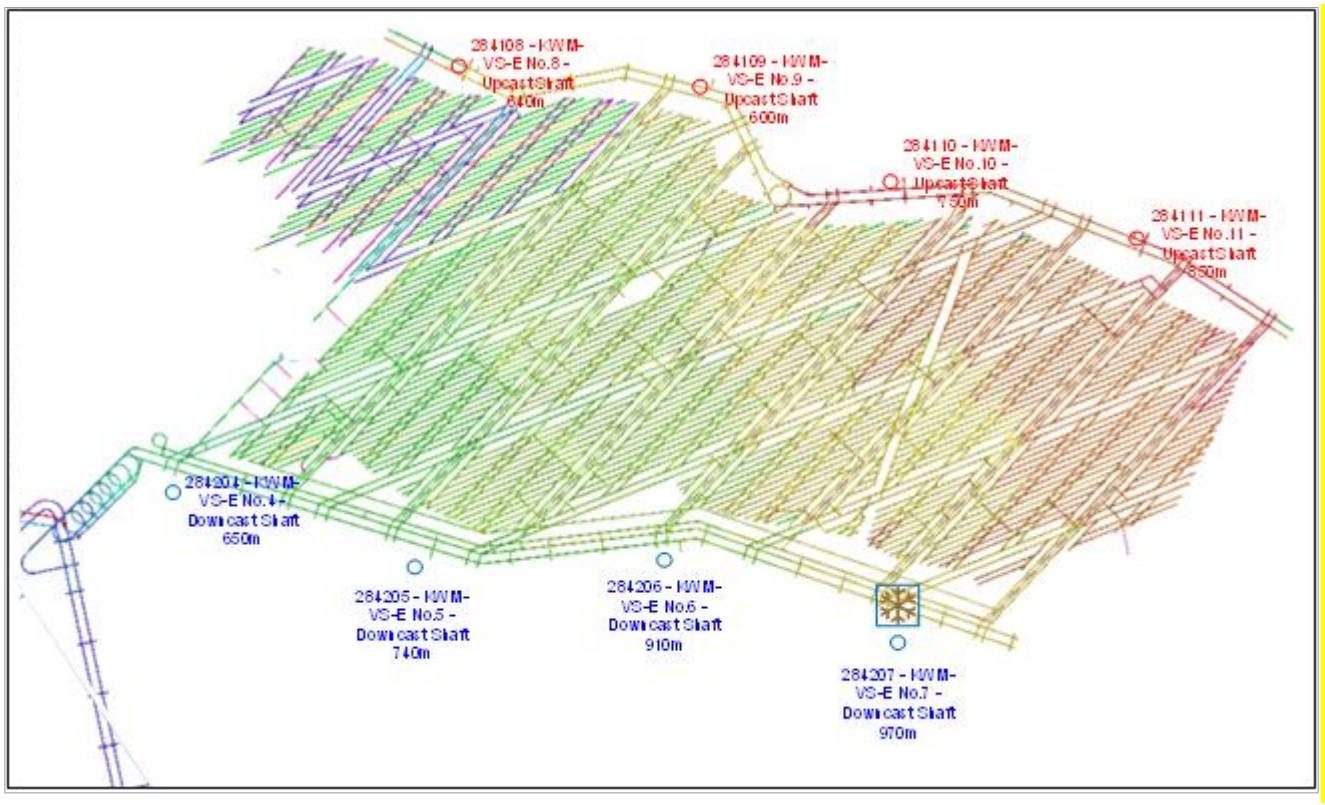
16.14.3.1 Kakula West (East) Ventilation strategy

The orebody will be accessed from surface via the Kakula West twin-decline system, which functions as a shared intake for both the West and East sections as shown in Figure 16.111. Fresh air will be supplied through the declines and four 6.0 mØ intake ventilation shafts positioned primarily south of the orebody. Guided by ventilation controls, airflow will be directed through the perimeter drives to ventilate the development and stoping areas. Ore extraction will be carried out using cut-and-fill mining methods.

Figure 16.111 illustrates the mine layout, showing the ventilation shaft locations and associated refrigeration facilities at the downcast shaft. The ventilation system is designed to deliver fresh air via the declines and intake shafts, one of which will be equipped with coolers, situated south of the orebody. Ventilation air will then circulate through the perimeter drives, linking the northern and southern extents of the orebody. A refrigeration unit will be required for the final stages of the eastern, mining at the lowest part of the deposit

The overall ventilation quantity for this mine is determined by the need to dilute diesel fumes, remove heat (distribute cooling) and dilute blasting fumes during re-entry. It must be noted that the overriding factor here is to dilute diesel fumes. The model shows the primary ventilation requirements of 1,621 kg/s at the peak production rate of 2.4 Mtpa, and a maximum depth of 962 m.

Figure 16.111 Schematic - Kakula West (East) Mine layout with ventilation shaft locations



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16.14.4 Kakula West - East production

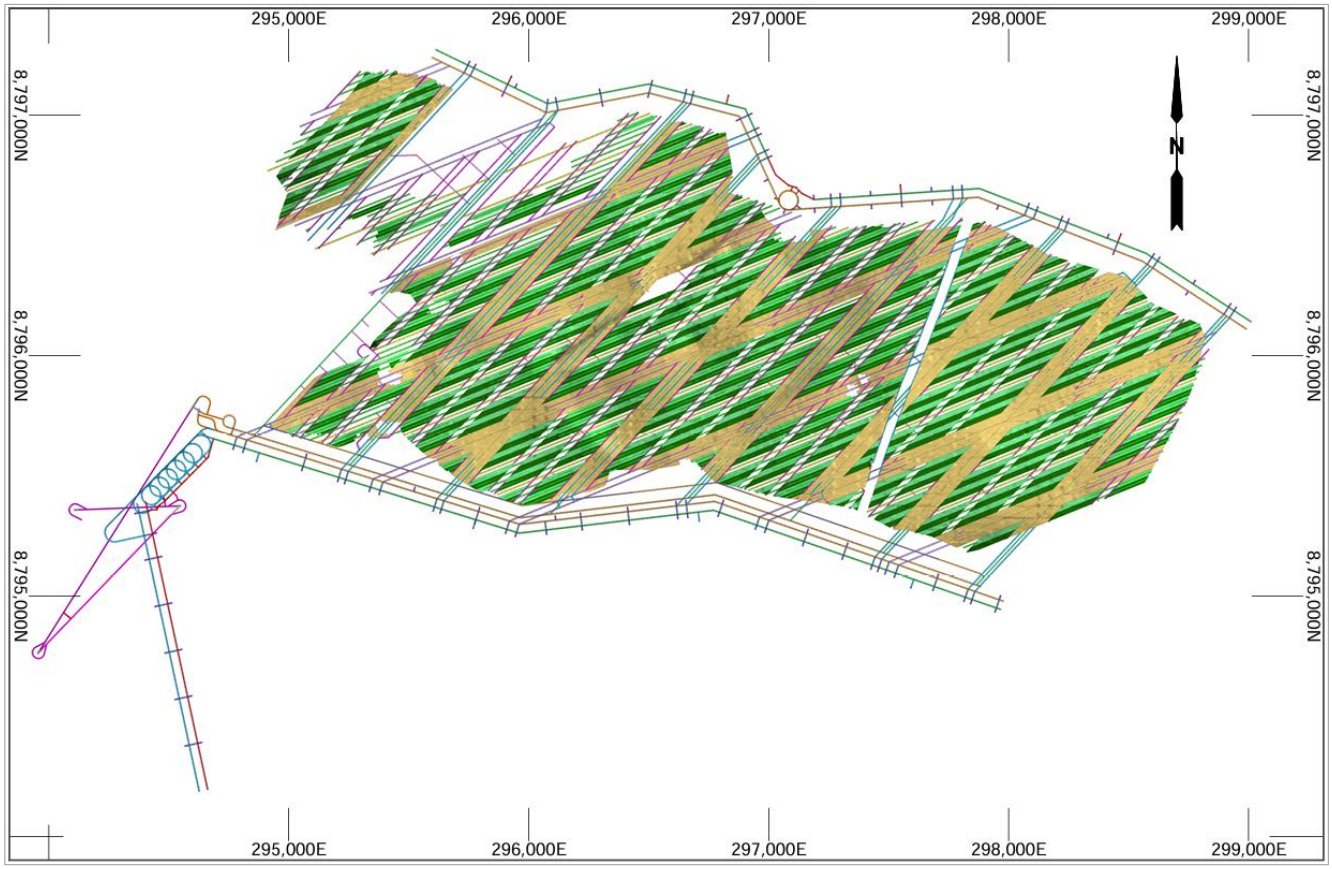
The production plan for the remainder of Kakula West-East spans 37 years to produce 52.5 million tonnes of ore and 1.3 million tonnes of copper. The production profile is described below in Table 16.61

Table 16.61 Kakula West-East production profile

Existing Kamoa Mine	Ore Tonnes (Mt)	Grade (%Cu)	Cu Metal (Mt)	Peak Production (Mtpa)	Steady Production (Mtpa)	Production Life (Yrs)
Kakula West - East	52.5	2.5	1.3	3.0	1.6	37

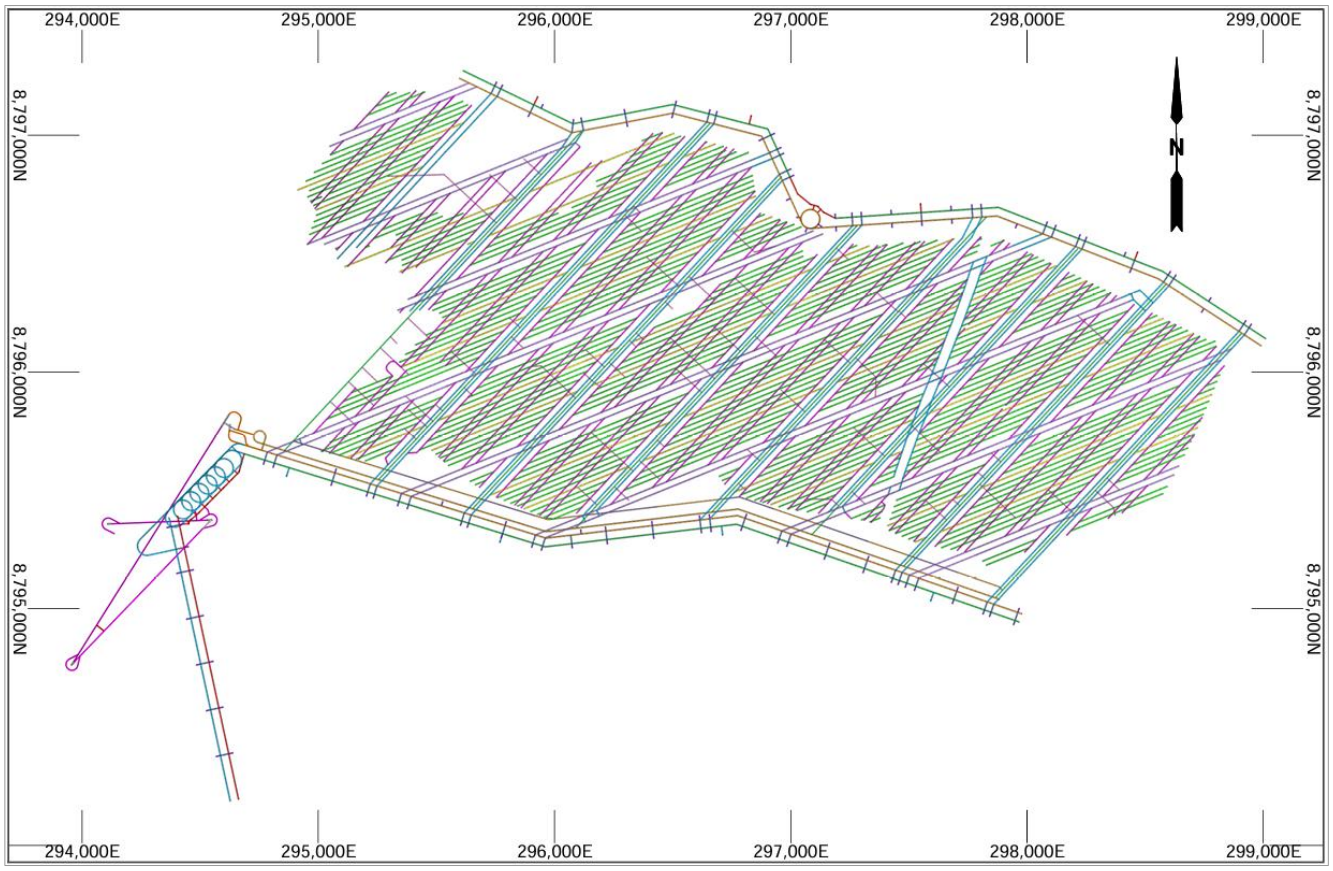
16.14.4.1 Production summary

Figure 16.112 Kakula West - East plan view



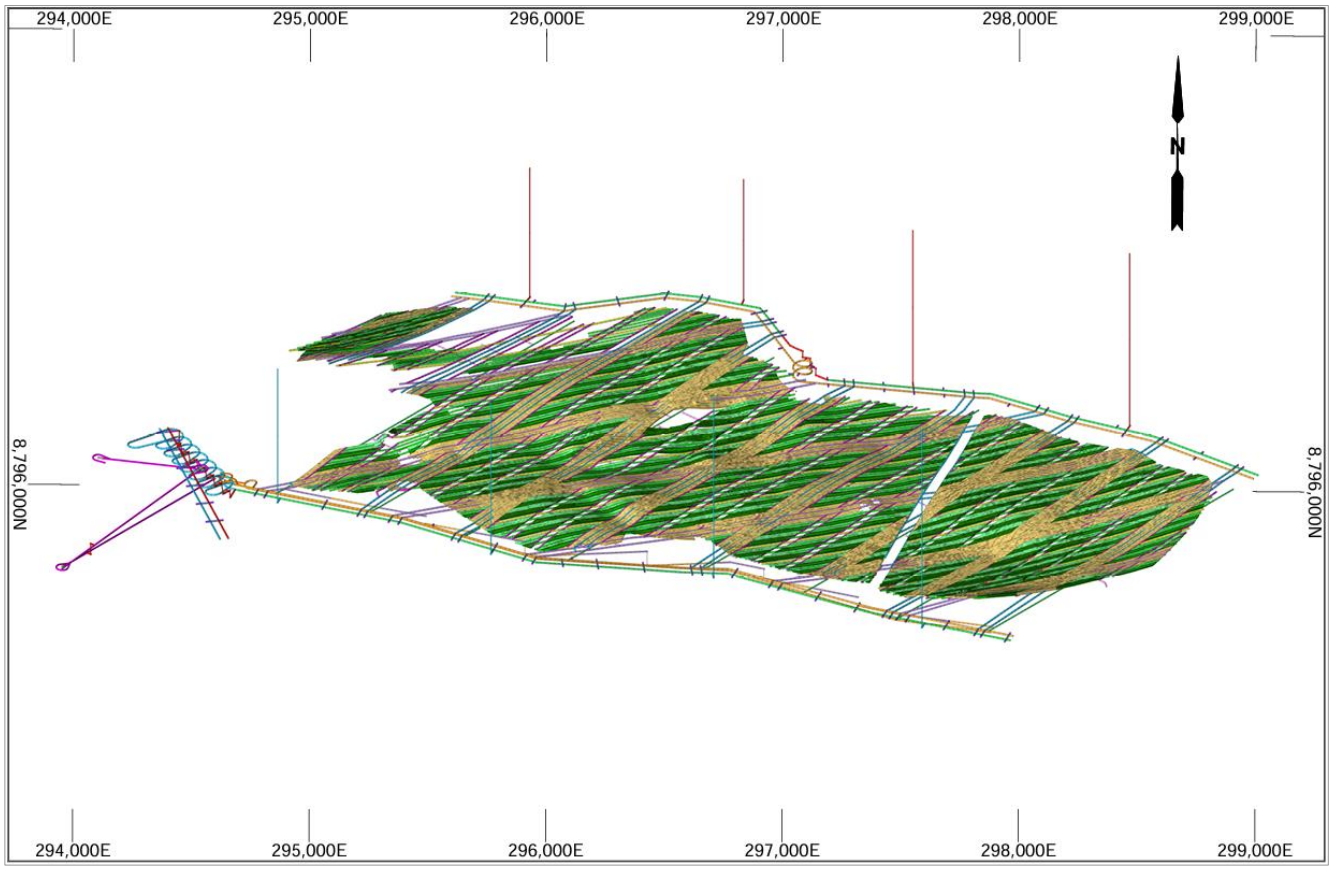
Source: AMC 2026

Figure 16.113 Kakula West - East Plan View (Drives only)



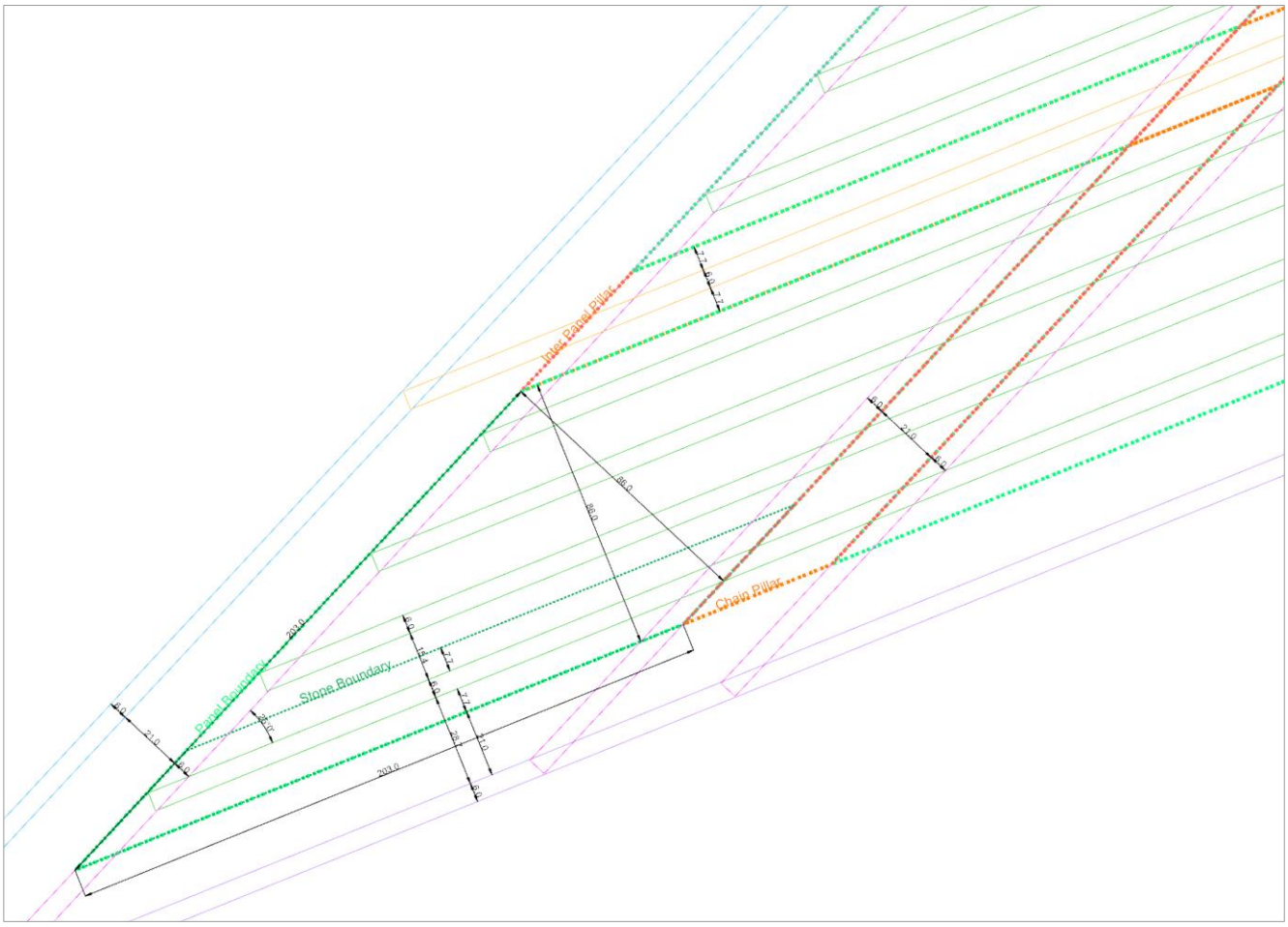
Source: AMC 2026

Figure 16.114 Kakula West - East isometric view



Source: AMC 2026

Figure 16.115 Typical layout and dimensions for a panel within a mining block



Source: AMC 2026

The stope design for this project employs longhole (LH) drilling techniques, enabling mining operations to proceed on both the left and right sides of the ore drive. The maximum stoping height is restricted to 7.5 metres to ensure operational safety and maintain ground control standards. The production process is governed by a series of operational rules that include dependencies applying access requirements, ventilation and drainage connectivity. The panel and stope sequencing follows Beck (2025) recommendations, utilizing a primary-secondary approach within each panel and incorporating paste backfill to maintain stope integrity.

Backfilling strategies are tailored to the type of stope: primary stopes can be filled from both adjacent drives and ends, offering greater flexibility in scheduling and logistics, whereas secondary stopes are only accessible for filling from the ends and may require staged filling procedures. Special attention is required for the interaction between backfill operations and the drainage horizon, necessitating active management and specific risk mitigation measures to ensure operational integrity and minimize potential issues.

The task rates below are applied to the tasks within the schedule. Production drilling is based on a P50 benchmark figure, whilst loader and backfill rates have been provided by the client. Individual stope production rate is planned to match the loading rate over the course of a month. Cable bolting has been conservatively applied based on experience. Cable bolting and Production drilling rates were flexed in the

schedule, testing increased activity rates through multi-resolution work sites in stopes. This is possible due to the long extent of ore drives, however not recommended in the base plan. Changing either of these factors does not have a significant impact on primary and secondary production.

Table 16.62 Activity and resource rates

Activity	Rate
Cable bolting (all types)	120 m/day
Production drilling	180 m/day
Loader	800 t/day
Stope Production Rate	35,0000 t/month
Paste backfill	800 m^3/day

16.14.4.2 Modifying factors

Dilution and recovery is summarized in Table 16.63 below. Dilution and recovery consists of two components. A baseline application of dilution (10%) and recovery (5%), which is considered typical for this level of study for stopes.

In addition, an allowance made for compensating the design inaccuracy on the floor of the stopes. Comparison of the design vs operationally achievable shapes show variation which will need amending and further clarification with future design work.

A review of this inaccuracy shows a range in the applicable dilution and recoveries that apply across the mine. The spatial average across all test locations equates to a 18.5% variation. Detailed design, through additional study work or during implementation will prioritize additional dilution over reduced recovery to maximize total recovered Cu. The dilution penalty is estimated as an average of an additional 8.5% material mined to create an operational floor, with a reduction in recovery estimated to be a further 10% on baseline where material is below an operational floor or warrants to much dilution to ensure it is accessible for bogging.

Dilution within the mine plan is allocated 0% Cu Grade. Variation in grade near the minable ore boundary could result in additional recovered material, providing upside to dilution at an operational level.

Pillar extraction within the mine plan has been further refined by an overall 30% extraction factor applied to the design shapes. They are further penalized due to the anticipated deteriorated ground conditions at the end of mine life. The penalty is an additional dilution of 5% and reduced recovery by 5%.

Table 16.63 Dilution and recovery factors

Category	Dilution factor	Recovery factor
Lateral development	1.05	0.95
Vertical development	1	1
Primary stoping	1.185	0.85
Secondary stoping	1.185	0.85
Pillar extraction	1.235	0.8

At a conceptual level, the mine layout offers a theoretical extraction (based on 2D plan area) of stoping and pillars is shown in Table 16.64 below.

Table 16.64 Theoretical 2D extraction ratio per complete block

Category	Recovered area m^2 (plan view)	Percentage of total (%)
Recovered	193,000	58.0%
Non-Recovered	140,000	42.0%
Total	**333,000**	**-**

16.14.4.3 Production profile

The current production timeframe for an individual stope and panel extraction sequence represents a significant bottleneck. While revising this sequence may lead to improved outcomes, further geotechnical modelling is required to justify any changes. A key challenge with the existing sequence is the uneven release of stope tonnes, resulting from the panel extraction order. When this is combined with development constraints and the relatively small stope tonnes accessible from available drawpoints, the outcome is an erratic production profile. These factors suggest that a lower, more consistent production rate may better suit the characteristics of this orebody.

The commencement of ore production is limited by both block and panel mining rules, as well as the necessity to establish dedicated return air infrastructure. These measures are essential to ensure that sufficient ventilation is available to support simultaneous development and production activities in close proximity. Additionally, the drainage horizon must be connected to the area below the block for effective dewatering. It is assumed that the pump station will be operational to support mining activities by the time production begins.

16.14.5 Kakula West - East development and infrastructure

The KW-E development design has been revised from the PEA to improve orebody access and overall design. Capital development is strategically placed to allow flexible access and ventilation, following geotechnical recommendations for reliable operations.

The mine plan now standardizes mining profiles, with detailed engineering and equipment selection planned for future work. There may be opportunities to reduce drive sizes, potentially lowering excavation costs and improving the extraction ratio.

Capital infrastructure estimates are based on access drives and include workshops, crib rooms, service areas, electrical bays, dewatering stations, and magazines. These will be refined in future studies.

Mine water is expected to cause operational delays and added costs due to pumping, flooding, and roadway degradation. A dewatering horizon is planned to divert water away from active drives, graded at 2% for improved safety and access.

16.14.5.1 Basis of design

The basis of design for development is provided in the accompanying table. The maximum development grade shall be limited to 15%; however, non-critical areas may exceed this limit but must be smoothed during detailed design. Wherever feasible, a minimum grade of 2% should be maintained throughout the project. It is important to note that ongoing updates to the orebody model may impact gradients, and these should be reviewed as geological understanding evolves.

Schedule rates are generally aligned with P50 benchmark figures. Notable deviations include the campaign jumbo rate, which is below the 50th percentile, whereas critical on-orebody development is positioned at the 75th percentile.

Table 16.65 Excavation profiles

Description	Profile	Width (m)	Height (m)	Radius (m)	Gradient / Dip	Schedule Rate (m/mo)
Main Access	Arched	6	6	1.5	Variable	90*
Drain Drive	Arched	5	5.5	2.5	Variable	60
Fresh Air Way	Arched	6	6	1.5	Variable	40
Inter Panel Pillar Drive	Arched	6	6	1.5	Variable	60
Ledge Drive	Square	6	6	1.5	Variable	60
Ore Drive	Arched	6	6	1.5	Variable	60
Strike Perimeter Drive	Arched	6	6	1.5	Variable	90
Cross Cut Perimeter	Arched	6	6	1.5	Variable	90
Return Air Way	Arched	6	5.5	1.5	Variable	40
Strike Drive	Arched	6	6	1.5	Variable	60
Stockpile	Arched	6	6	1.5	1:50	40**
Capital Cross Cut	Arched	6	6	1.5	1:50	40**
Operating Cross Cut	Arched	6	6	1.5	Variable	60
Return Air Raise	Circle (Raisebore)	4.5 ⌀	-	2.25	90°	-
Fresh Air Raise	Circle (Raisebore)	4.5 ⌀	-	2.25	90°	-

Note: * Adjusted to 150m/mo for campaign mine access
** Adjusted to 90m/mo for campaign mine access

16.14.5.2 Development mining and sequence profile

Significant upfront capital is needed due to orebody depth before access can be achieved. To enhance project NPV, campaign development is undertaken until the orebody is reached, typically using a skilled contractor for higher single-heading development rates.

The ramp-up of development depends on available headings and is limited to around eight jumbos per year to avoid excessive advance mining ahead of the stoping front. This limits the timeframe development is open, reducing the potential for considerable rehabilitation requirements.

Infrastructure costs are estimated by applying a factor to designed capital access meters, which include crosscuts and stockpiles. This means the calculated meters are exclusively for mine infrastructure. Access development within the orebody footprint, where mining occurs, incorporates factors for stockpiles, local electrical bays, refuge bays, as well as infrastructure components.

The development sequence adheres to standard mining practices, requiring turnouts as progression continues. Mining activities focus first on providing access and supporting development for operations to ensure consistent operations. Development turnouts also account for cable bolting at intersections.

16.14.5.3 Infrastructure requirements

Materials handling

Stope and development materials are transported either directly to trucks or to stockpiles, with subsequent rehandling to trucks as required. Truck haulage is executed via Perimeter Strike (PSK) or Perimeter Cross Cut (PXC) drives leading to the Southern Access drive. Transport through the PSK and PXC drives is comparatively slower due to their alignment with orebody geometry, which facilitates optimal take-offs in ore. Once on the Southern Access drive, standard gradient haul routes enable higher speed and efficiency for trucking operations toward the central shaft.

Stockpile development meters are factored into the design of PSK, PXC, and Access drives. On Surface is it expected that the waste dump is located close to the portal. The ore ROM should also be located close to the portal, minimizing impact to underground haul trucks.

Further studies will examine material handling trade-offs in greater detail. Preliminary assessment indicates that a haulage shaft is likely the most suitable solution for the KW-E deposit, as it would enhance both material transportation and personnel access. Additionally, positioning the shaft at an elevated location can mitigate risks related to capital infrastructure flooding in case of power outages.

Escapeways

Escape ways are planned for by being factored in the spiral decline design within the RAR and RAD development. These will be installed as the decline progresses. Mine development within the orebody is horizontal and offers two means of egress back to these escape ways. Schedule priority on establishing perimeter development aids ability to egress multiple ways. KW-E and KW-W should be linked via lateral development enabling multiple means of egress from the mine in the event of an emergency.

Refuge chamber cuddies are not included in the planned development but will be placed near work areas using available drives. Minor rehabilitation may be needed for previously used stockpiles, but this is outside the study scope. Stench gas will be positioned at all intakes and may be necessary at various parts of the mine to ensure timely exposure in the event of an emergency.

Ventilation and cooling requirement

Primary ventilation is delivered from the south and returns via northern infrastructure. Fresh air currently enters through four 4.5 m diameter rises and decline connections on the southern perimeter. Return air is extracted via four 4.5 m diameter rises. The twin accesses are advanced in tandem, allowing primary ventilation to progress with development and reducing reliance on secondary systems until main connections are in place. Initially, the first full-depth FAR for KW-E will serve as a return air rise until the northern RAR is completed. Once the first rise from surface is operational, the stepped RAR on the decline will be converted to a FAR, increasing fresh air supply and maintaining decline air velocity below 6 m/s.

Drainage drives may be utilized for ventilation if required, though further work is needed to confirm appropriate connections. The main drainage horizon trunk is has factored development within the schedule to accommodate ventilation links to support the long single heading.

Secondary ventilation will primarily run from the south-west towards the north-east, with fans positioned close to fresh air sources. External perimeters, Operating Cross Cuts (XCO), Ledges (LDG), and Inter Panel Drives (IPD) will be fully developed prior to stoping. This approach aims to provide effective secondary ventilation during both development and production phases.

Water management

Effective water management at KW-E is a significant operational challenge that must be addressed to sustain ongoing mining activities. The mine is hydraulically connected to surrounding aquifers primarily through the West Skarp Fault and associated sub-parallel faults. Previous groundwater studies indicate that inflows could reach a peak of approximately 4,500 litres per second, underscoring the scale of the issue for a horizontally extensive orebody such as KW-E.

To manage this groundwater influx, a dedicated drainage horizon has been implemented within the mine plan. This horizon has been strategically designed to intercept water from known low points, where pooling will occur according to the current development layout. Targeted development provides access for dewatering at these key locations. As a greater understanding of orebody knowledge becomes available, additional drives may be required to cater provide dewatering ability through drain holes.

The drainage horizon will be linked to the production footprint via large diameter boreholes, expected to be drilled with an In The Hole Hammer (ITH) drill rig. These dewatering holes vary in length and can be >60 m in length due the differential between orebody gradient and the drainage horizon. These large diameter holes present a safety hazard to operators and machinery. The open hole hazard as well as suction hazard when blocked must be managed. Barricading, SWPs, education, hole strainers and covers must be employed to mitigate risk of access to the top of the hole.

The direction of drainage is influenced by the dip of the orebody, but it generally trends towards the east. The constructed drainage horizon is designed to direct water south and east, ultimately channeling it towards the lowest point on the southern access. The development of declining headings is expected to be extensive, with development rates bring impacted if significant volumes of water are intersected during excavation.

As hydrogeological understanding of the area improves it may become necessary to undertake predevelopment activities and proactive dewatering boreholes via the drainage horizon in advance of the mining front. before advancing further mine development. These measures aim to minimize water ingress into mine production areas and maintain safe and efficient operations.

Significant development is anticipated in declining headings. Development rates may be impacted by excessive pooling water at the development face. As hydrogeological knowledge improves, predevelopment of the drainage horizon and dewatering boreholes could be necessary to reduce water inflows in the operating mine.

Figure 16.116 KW-E showing primary accesses and drainage horizon under the orebody footprint. Arrows show drainage direction.

Figure 16.116 Drainage Horizon below the Kakula West-East orebody

Source: AMC 2026

Power requirement

The initial power supply for underground operations will be established via the return air drive in the twin decline, transmitting electricity directly from the surface down to the working areas. As the development of the spiral decline progresses, the power supply will be extended through service holes to the southern perimeter access drives, ensuring coverage across key operational zones.

Consideration must be given to the practical aspects of running the main trunk line within the first fresh air rise. This assessment should take into account the chosen installation method, the required level of structural support, and any additional usage needs that may arise. The overarching objective is to provide a direct electrical feed from the surface to approximately the 650 metre level (mL), thereby establishing a ring main. This configuration will help to minimize reliance on electrical supply lines routed within development drives, enhancing both efficiency and reliability of the underground power distribution network.

Other

Compressed air will be distributed throughout the mine to facilitate and support various operational activities. Air compressors will be located underground, close to return air to mitigate risks in the event of a fire. These may utilize connections to the drainage level with return air being drawn down from the main production horizon, across to the northern perimeter.

To minimize the transit time associated with blasting activities, underground magazines will be utilized. Initially, these magazines will be situated on the Southern Perimeter of the mine. Once return air flow is established in the northern areas, the magazines will be relocated to the Northern Perimeter.

Fuel bays will be positioned adjacent to return air pathways and will be supplied via a fuel dropper system, ensuring efficient fuel delivery for underground operations.

Service bays and laydown areas will be established near the active working zones. This proximity will enable timely access to necessary supplies and support ongoing mining activities.

Communications infrastructure, IT systems, underground offices, and crib rooms will be located close to the base of the decline. This arrangement will support the initial years of mine production and provide a central hub for operations throughout the mine's lifespan. Consideration should be given in future to pairing these facilities with a shaft hoist base or access point, with modifications made as necessary to suit operational requirements.

Fixed infrastructure

To support the ongoing underground mining operations, dedicated machinery workshops will be required. These workshops will include both heavy and light vehicles.

The heavy machinery workshops will be designed to service production and development drills, as well as haulage equipment including loaders and trucks. While the majority of truck servicing is expected to occur on the surface, particularly during the ramp-up phase of operations, the underground workshops will provide critical support for these and other heavy machinery required for daily mining activities, especially towards the later stages of mine life.

Light vehicle workshops will be included in the base plan to do critical servicing, breakdowns and daily’s. Longer period serving and major rebuilds / repairs will be conducted on the surface.

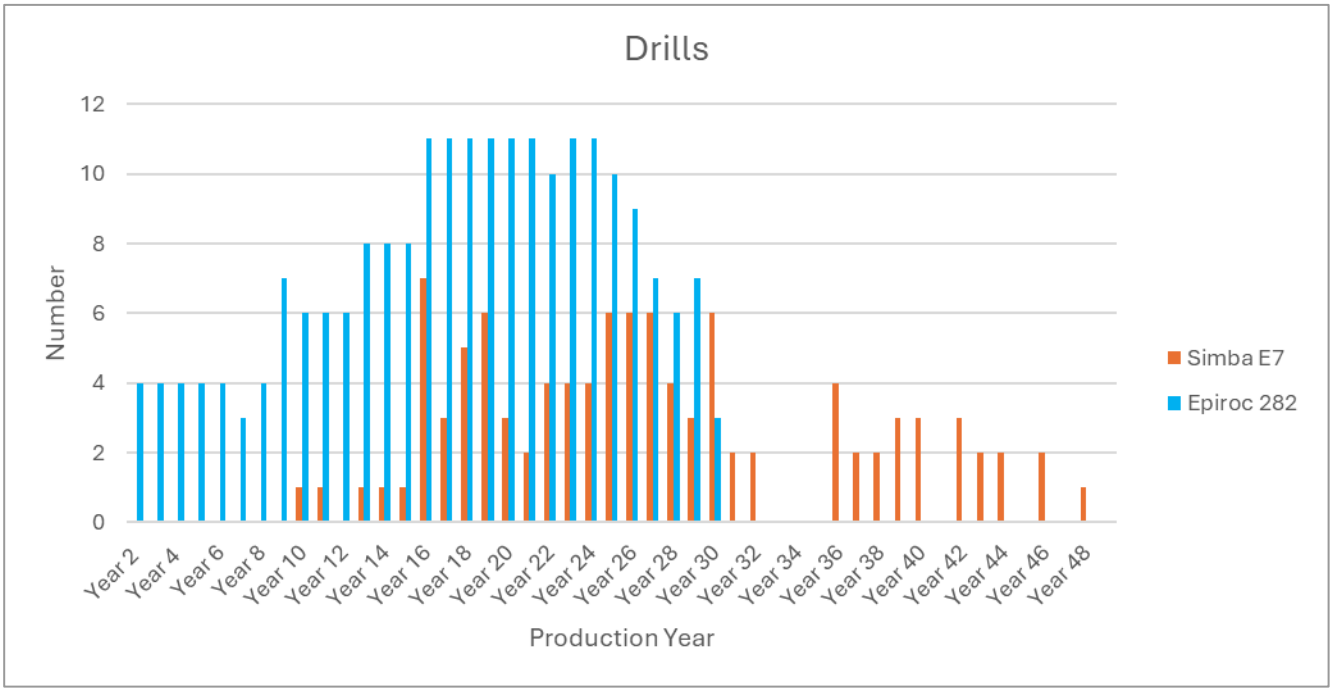
The exact sizing of these workshops has yet to be determined. However, the mine development plan has taken into account the need for flexibility, allowing for future revisions and expansions of these facilities as operational requirements evolve.

16.14.6 Kakula West - East equipment requirements

16.14.6.1 Mobile equipment fleet

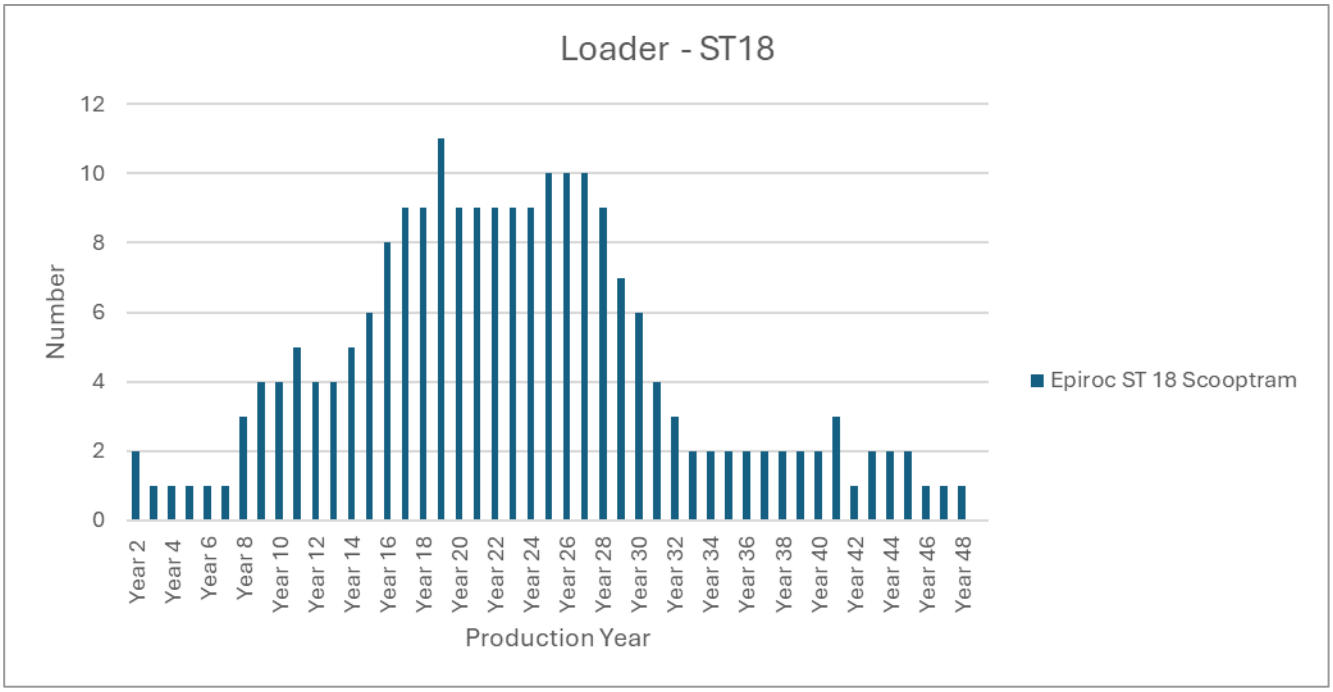
The following Figure 16.117, Figure 16.118 and Figure 16.119 the planned equipment requirements for drills, loaders and trucks respectively.

Figure 16.117 Kakula West - East drill equipment schedule



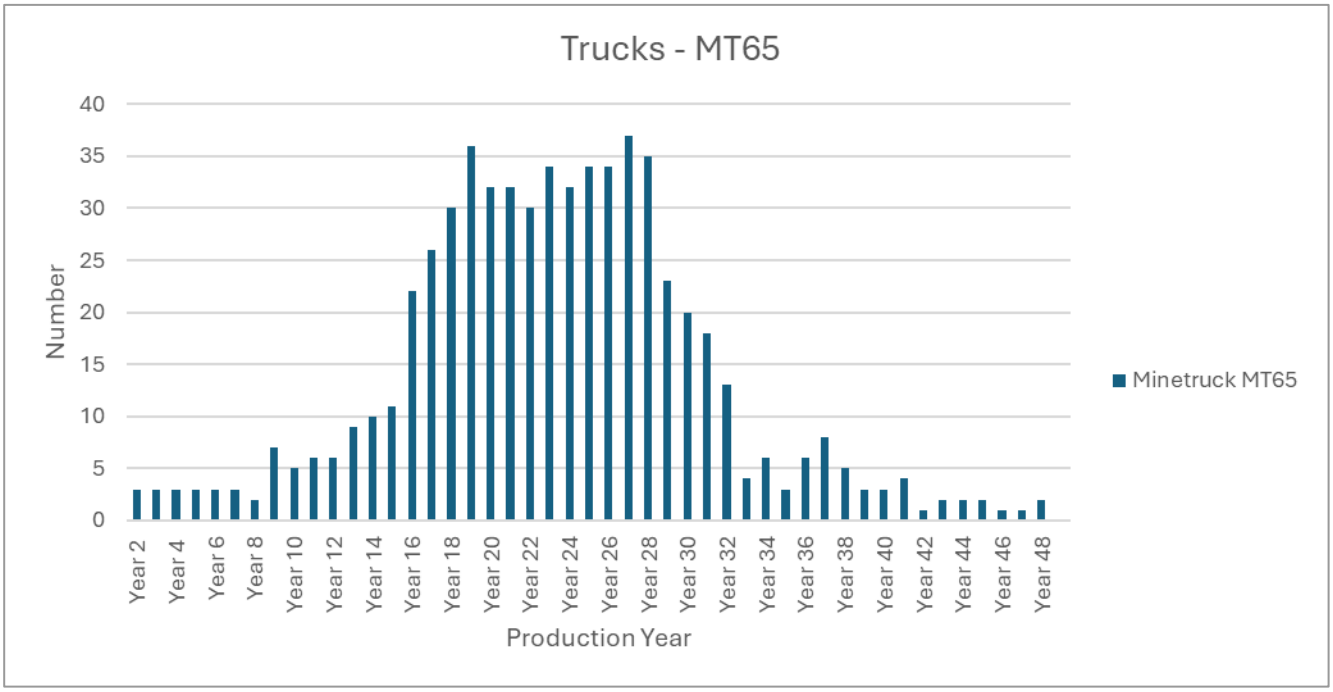
Source: AMC 2026

Figure 16.118 Kakula West - East loader equipment schedule



Source: AMC 2026

Figure 16.119 Kakula West - East truck equipment schedule



Source: AMC 2026

16.14.6.2 Fixed equipment

The fixed infrastructure for the mine will be consistent with standard requirements for a stoping operation. Due to the lateral extent of the orebody, there may be a need to relocate or replace infrastructure at various stages throughout the mine's life to maintain operational productivity.

Further assessment is required for materials handling and integration between KW-E and KW-W. Future studies should address these aspects, as alignment between the two mines is anticipated to provide operational synergies.

16.14.7 Recommendations and next steps

To ensure effective project advancement, the following technical and planning actions are recommended:

- Integration of KW-E and KW-W operations should be investigated.
- The layouts for mines and stopes require review, including adjustment of block patterns to better align with orebody geometry and removal of steep grades from production plans.
- Stope design needs further refinement, and unnecessary planning factors should be eliminated.
- Enhancements to the materials handling system must be tested to improve efficiency.
- Alternatives for dewatering horizons should be assessed with the aim of minimizing capital development requirements.
- There is a need to increase the level of detail in pillar recovery planning.
- Geotechnical modelling of mine plans should also be conducted.
- Lastly, evaluation of potential access via the West Skarp fault and possible integration with Kakula East infrastructure is recommended.

16.15 Kakula West – West

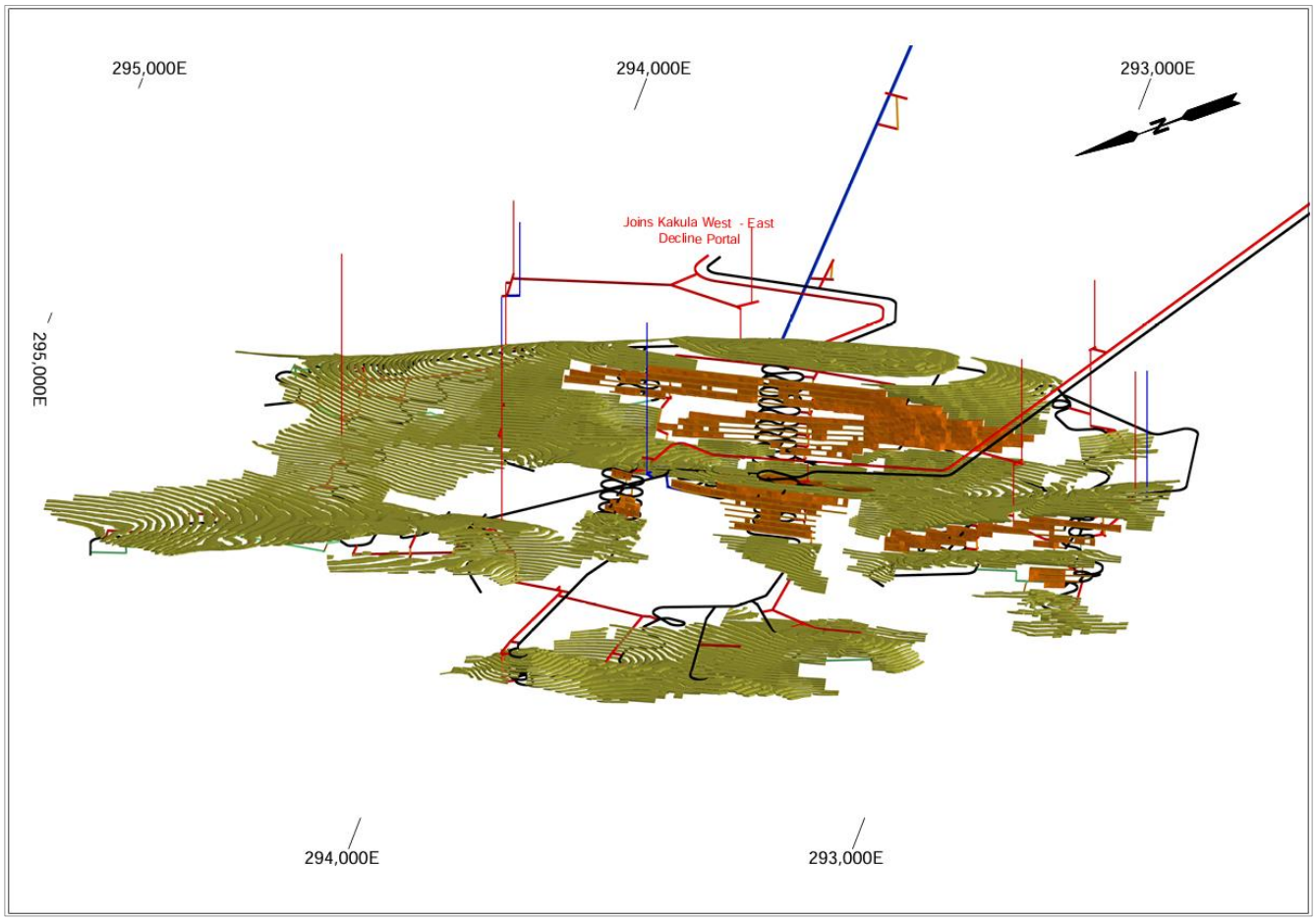
16.15.1 Summary of relevant information

The Kakula West Orebody forms the western extension of the ultrahigh-grade Kakula Trend within the Kamoa-Kakula Copper Complex. The deposit is a large stratiform copper deposit, sharing the same distinctive stratiform style, mineralogy, and geometry as the main Kakula orebody. The deposit is tabular, with the orebody dip ranging between 0 – 58°, with varying thicknesses up to approximately thirteen metres. The western extent of the Kakula West deposit shows a much greater variation in orebody dip than the eastern extent, representing the extensive folding and undulating geometry of the orebody with offsets. Subtle disruptions in ore continuity are indicated in the block model, particularly in the steeper dipping areas of the orebody. This results in a highly variable depth of the orebody, with a maximum depth of 800 metres below surface. The ground conditions are described as poor to fair. The mineralized zone extends for approximately three kilometres, encompassing both the east and west extents of the Kakula West mining areas.

The mining methods planned for Kakula West – West are a combination of cut and fill (C&F) mining and long hole stoping mining (LHS), with both methods applying a bottom-up mining sequence. C&F mining is planned in the shallower dipping areas of the orebody, and LHS mining where the orebody steepens. Consolidated backfill will be required in both mining methods. (See mining method description in Section 16.4.3).

The overall geometry of the Kakula West - West orebody and the planned locations of the C&F and LHS mining areas are shown in Figure 16.120. Both C&F and LHS methods require extensive capital development and infrastructure in waste material, and for these activities to be completed ahead of ore production to enable safe and efficient extraction. The extent of the design of lateral capital waste development allows establishment of the primary ventilation circuit in addition to the potential of multiple concurrent production areas.

Figure 16.120 Kakula West - West mining methods



Source: AMC 2026

C&F is a highly controlled, sequential, supported mining method. C&F mining commences once the decline reaches the level access elevation of the orebody, usually midway along its strike length. Figure 16.120 shows a schematic of the C&F mining method. C&F is an overhand mining method and the extraction sequence begins with the lowest 5.0 m high lift. Then each subsequent lift requires the back of the Level access to be slashed down to reach the next lift. The number of lifts between levels accesses is variable depending on the geometry of the orebody in each location.

C&F mining will be a combination of single pass or multi-pass C&F, depending on the orebody widths from footwall (FW) to hangingwall (HW). The mining begins by driving the level access to the orebody contact of Lift 1 (Figure 16.120AMC 2026). The primary drift will be mined at a 5.0 m x 5.0 m profile along strike to the extent of the localized stoping area. The drift will then either be slashed out on retreat if required, and then the drift will be backfilled. Depending on orebody thickness, after the fill has cured, a secondary drift will be developed parallel to the primary drift, with fill on one side and ore on the other side of the drift. Once the final placement of fill has cured, the level access will be slashed down to reach the next lift and the process will be repeated for the remaining lifts of multi-pass C&F.

16.15.2 Dewatering for Kakula West - West

See 16.14.2 Hydrogeological Setting for Kakula West - East.

16.15.3 Kakula West-West Ventilation

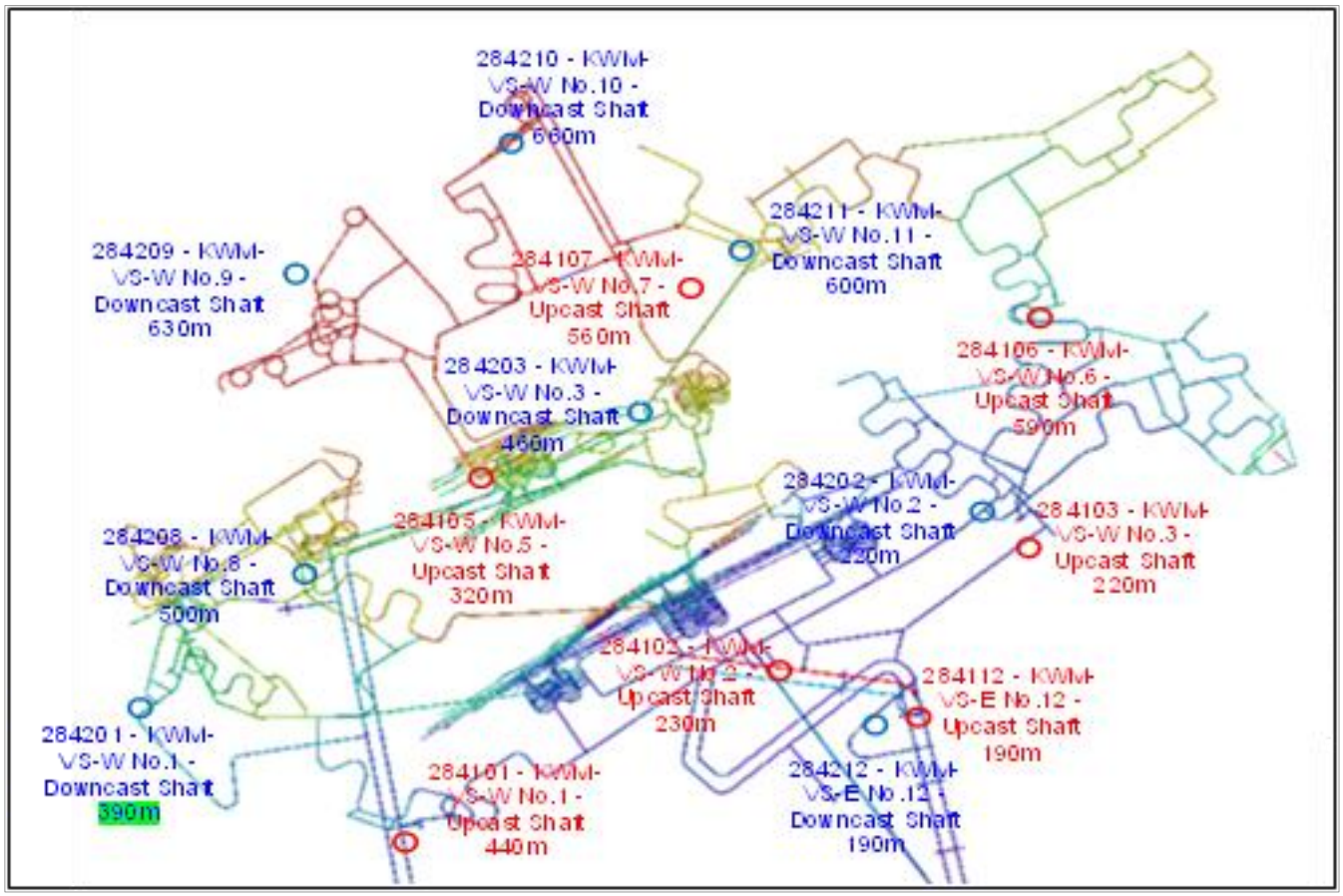
Kakula West (West) Ventilation strategy

The orebody will be accessed from surface via the two Kakula West twin declines. This decline will serve as a shared intake for Kakula West (East).

Figure 16.121 illustrates the mine layout and the positions of the ventilation shafts. The ventilation system is designed to supply fresh air through the declines and intake shafts distributed across the orebody. Guided by ventilation controls, airflow will move through the perimeter drives and ramp systems, where auxiliary fans will deliver it to the stoping and development faces. The air will then be directed into the return system and exhausted to surface via the main fan stations. Ore extraction will be carried out using cut-and-fill mining.

The model shows the primary ventilation requirements of 1,860 kg/s at the peak production rate of 2.4 Mtpa, and a maximum depth of 804 m. No refrigeration is required

Figure 16.121 Kakula West (West) Mine layout with ventilation shaft locations



Source: DRA Global 11/11/2025.

16.15.4 Kakula West - West Production

Based on the expected poor to fairground conditions, a conservative extraction approach was applied to both the Drift and fill and LHS mining methods. A 60% recovery factor was applied to the D&F stoping areas, allowing for significant pillars to remain for regional stability. Detailed geotechnical design will be required

in the LHS mining areas following further data collection for parameters to finalize the Hydraulic Radius (HR) and ELOS for stope design.

The production plan for the Kakula West-West spans 30 years to produce 32 million tonnes of ore and 1.2 million tonnes of copper. The production profile is described below in Table 16.66. The operation is expected to take 9 years to reach full capacity of around 2.2 Mtpa and will remain at this level for approximately 8 years and then steadily decline.

Table 16.66 Kakula West-West Production profile

Existing Kamoa Mine	Ore Tonnes (Mt)	Grade (%Cu)	Cu Metal (Mt)	Peak Production (Mtpa)	Steady Production (Mtpa)	Production Life (Yrs)
Kakula West - West	31.9	3.8	1.2	2.2	1.5	30

The parameters used for stope optimization are shown in Table 16.67 in Section 16.15.4.1. These stope optimization parameters are considered to be conservative.

16.15.4.1 Production summary

Production is defined by the two different mining methods, C&F and LHS. C&F production stopes are designed with a minimum height of 5.0 m and a maximum height of 7.5 m. The stope shapes are cut to a contour of 7.5 m heights with varying stope widths depending on the geometry of the orebody, resulting in a significant variation in the lateral widths of the stopes in this design phase. Due to the undulating nature of the orebody, significant decline development and related ventilation development is required to access the various C&F mining areas, although this development has the advantage of establishing multiple independent mining areas.

LHS stope design parameters are shown in Table 16.67 below. Due to the limited geotechnical guidance available, and considering the poor to fairground conditions, a conservative approach was applied to Mineable Shape Optimizer (MSO) parameters.

Table 16.67 Kakula West – West Stope Optimizer parameters

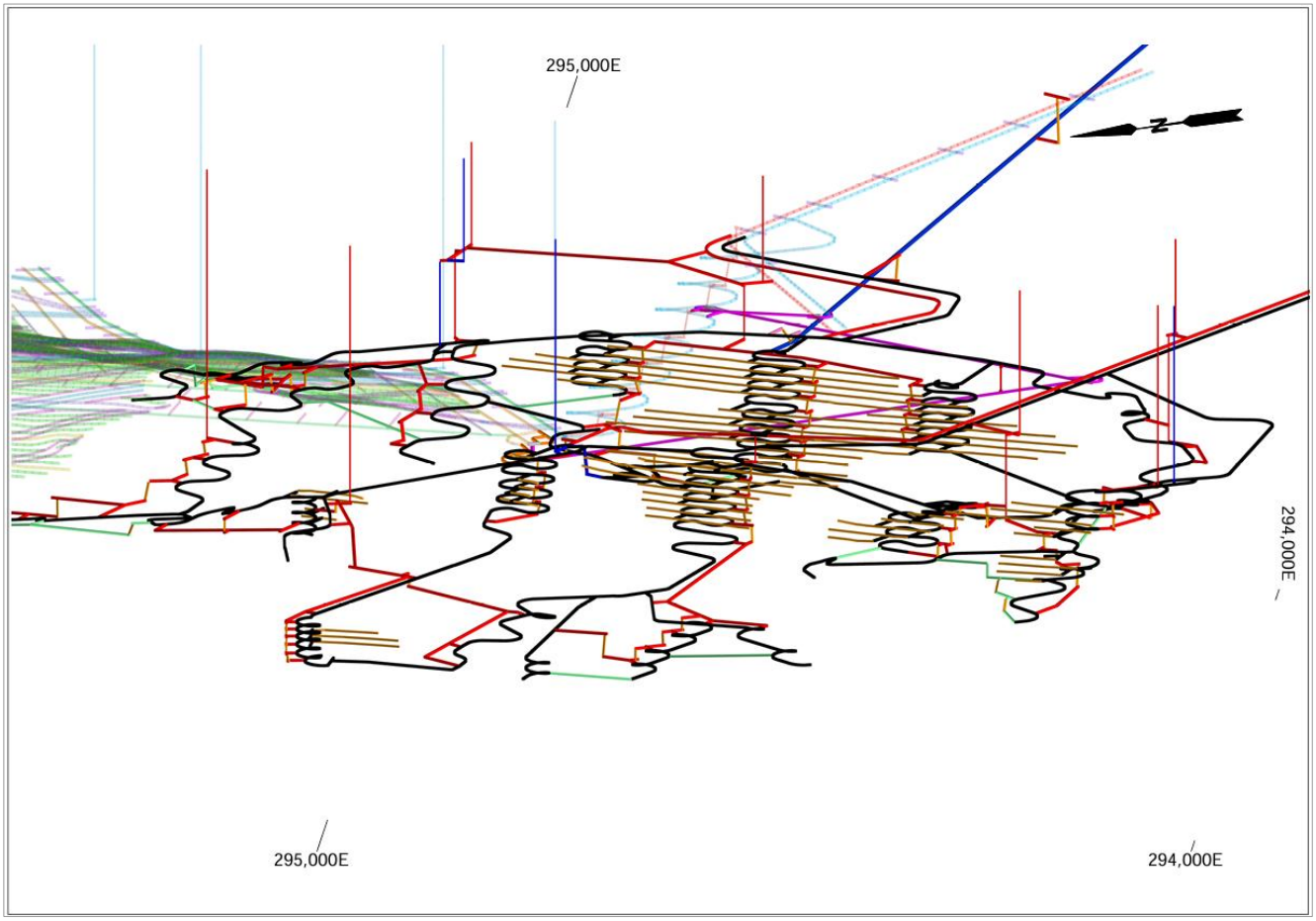
Item	Value	Justification
Level interval	15 m	Poor to fairground conditions
Longitudinal minimum mining width	4 m	Client Guidance
Longitudinal maximum mining width	100 m	To investigate max width of potential stopes
Longitudinal stope length	15 m	Poor to fairground conditions
Footwall dilution	Nil	Managed via modifying factors in schedule
Hangingwall dilution	Nil	Managed via modifying factors in schedule
Backfill method	Paste	Geotechnical considerations

The Kakula West - West equipment and task rates validated with the AMC SmartData™ benchmarking database are shown in Table 16.68.

Table 16.68 Kakula West – West Schedule equipment and task rates

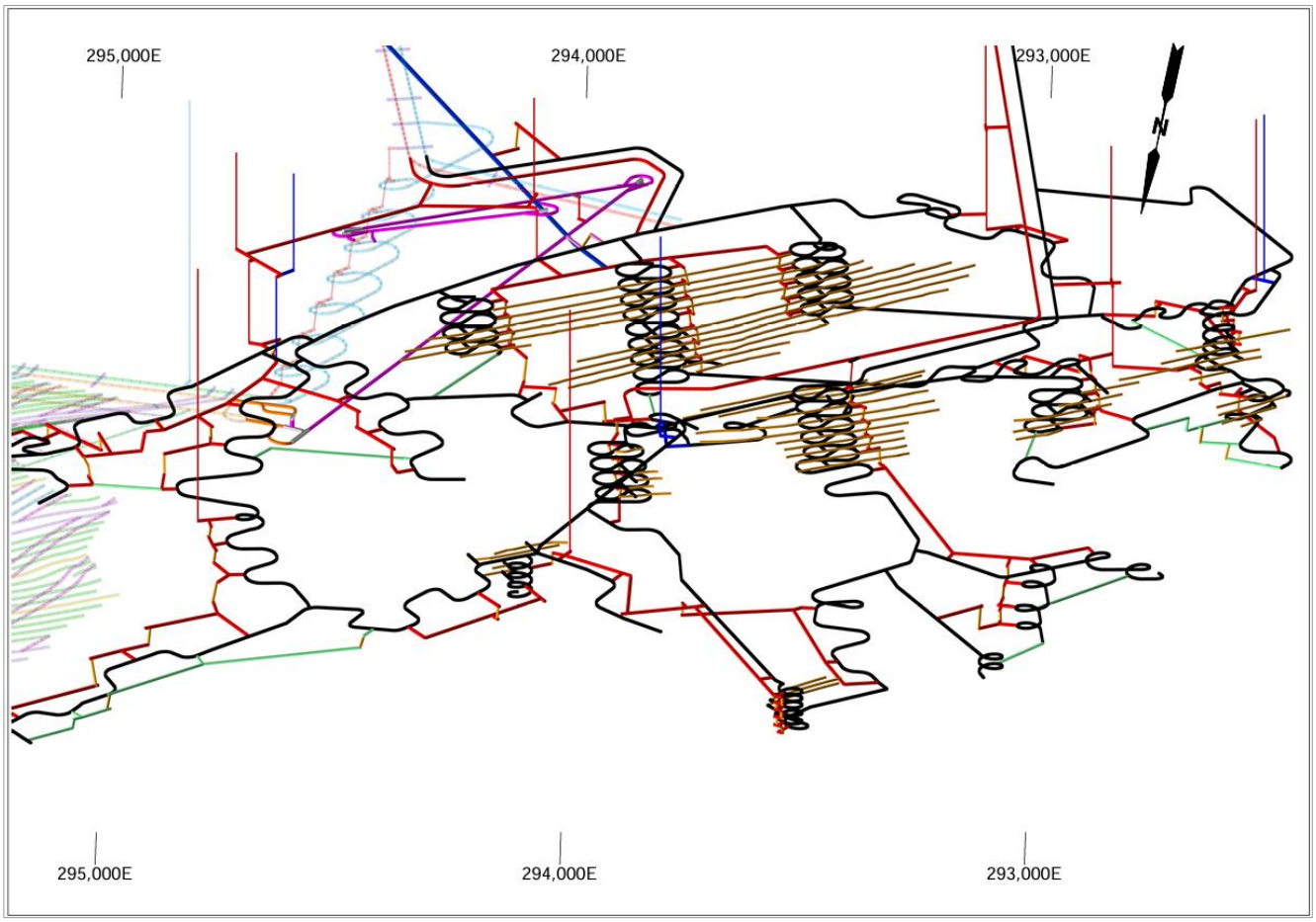
Equipment or Task	Rate
Jumbo	130 m/month
Production drill	135/day
Raisebore	2/day
Stoping	800/day
Cable bolter	150/day
D&F jumbo	9828/mo
Paste plant	4000/d

Figure 16.122 Kakula West - West and Kakula West - East isometric



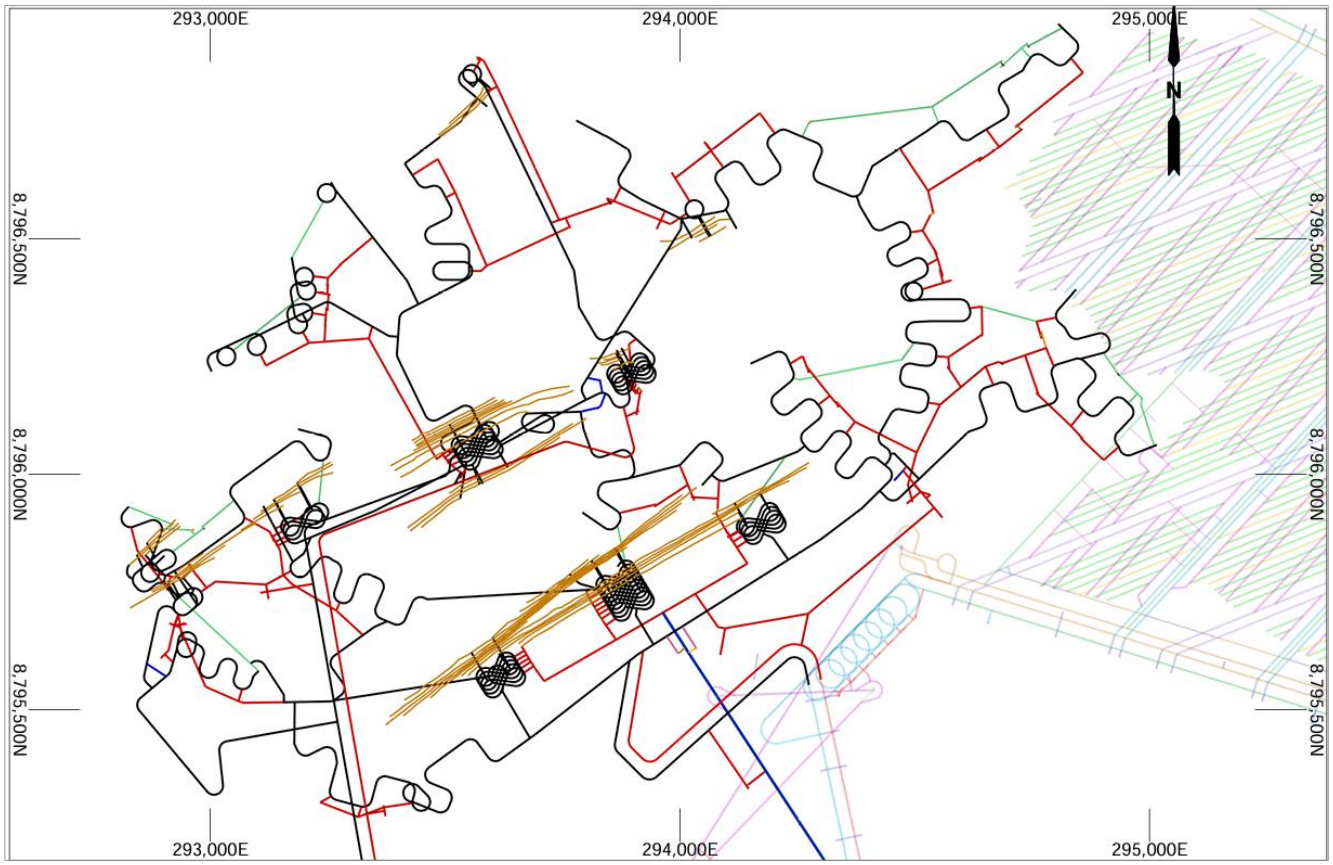
Source: AMC 2026

Figure 16.123 Kakula West - West isometric



Source: AMC 2026

Figure 16.124 Kakula West - West plan view



Source: AMC 2026

16.15.4.2 Modifying factors

The modifying factors applied to the Kakula West – West design activities are shown in Table 16.69. All C&F activities were limited to a 7.5 m height restriction, with a minimal amount of areas reaching this height.

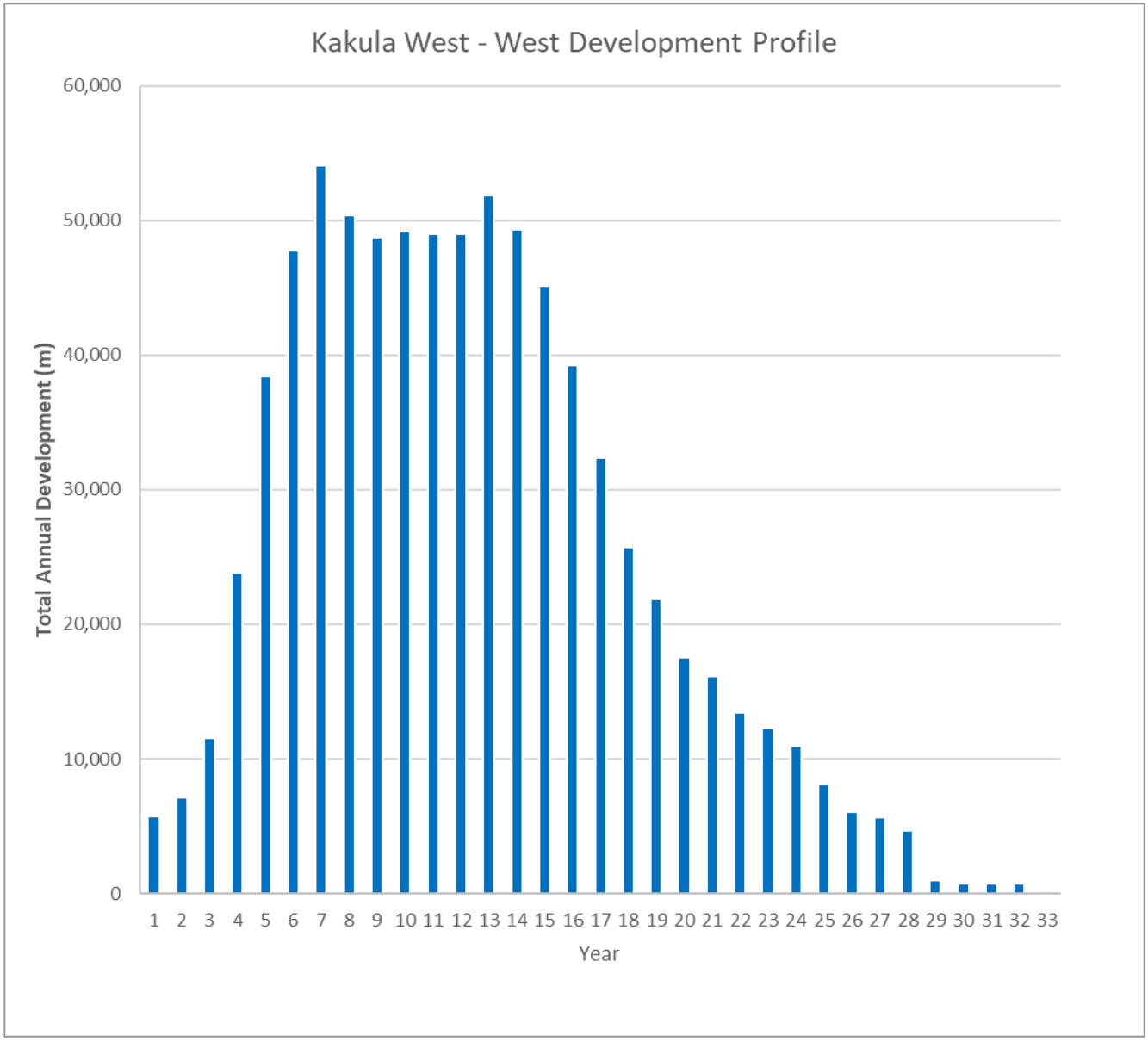
Table 16.69 Kakula West – West Modifying Factors

Activity	Dilution factor	Mining recovery factor
Horizontal development	1.05	0.95
Vertical development	1.0	1.0
Drift & Fill	1.1	0.65
Stoping	1.1	0.9

16.15.5 Kakula West - West development and infrastructure

The Kakula West – West advance profile is shown in shown in Figure 16.125.

Figure 16.125 Kakula West - West annual development advance profile



Source: AMC 2026

16.15.5.1 Basis of design

The design assumptions for both lateral and vertical mining tasks are shown in Table 16.70 below.

Table 16.70 Kakula West – West excavation profiles

Description	Profile	Width (m)	Height (m)	Gradient / Dip
Decline / Incline	Arched	6.0	6.0	Variable
Conveyor decline	Arched	8.0	6.0	1:6
Haulage drive	Arched	6.0	6.0	Variable
Major access	Arched	6.0	6.0	Variable
D&F ore drives	Arched	5.0	5.0	Flat
LHS ore drives	Square	4.5	4.5	Flat
Fresh air drives	Arched	6.0	6.0	Variable
Return air drives	Arched	6.0	6.0	Variable
Escapeway raise	Circle (Raisebore)	1.5 ⌀	-	70-90°
Ventilation raise	Rectangle (Longhole)	4.5	4.4	70-90°
	Circle (Raisebore)	3.5 ⌀	-	90°

16.15.5.2 Infrastructure requirements

The Kakula West – West mine will require a backfill plant on surface, and a conveyor system for transporting ore to the ROM and processing plant.

Materials handling

D&F mining requires a high number of working areas to be active simultaneously to achieve a reasonable total production rate. The combination of D&F mining and LHS mining increases the number of working areas required. This necessary dispersion of activities lends itself to a trucking method for materials handling.

Each mining domain will be accessed by a decline, and ore will be trucked to a central conveyor loading station. From there, all material will be conveyed to the surface and to the ROM.

Depending on future extensions to the mine, the option of a shaft hoist system may be considered later in the mine life as depth becomes an issue for the large trucking fleet.

Escapeways

The mine development design, shown in Figure 16.121 shows fresh air intakes in blue. These can double as escapeways and secondary egress. The design also includes a series of connection drives between declines that allow for multiple escape paths for secondary egress through the extensive decline system.

Water management

Dewatering will be undertaken by level sumps and decline sumps with secondary pumps reporting to several larger main pump stations. The primary pump stations connect directly to rising main pipes directly to the surface.

Power requirement

Power will be supplied underground via service dropper holes from the surface. These holes will be used to connect to an underground substation and reticulation for each operational area.

Other

Compressed air will be provided by several underground compressor stations and receivers that will assist with reticulation.

Fixed infrastructure

To support the ongoing underground mining operations, dedicated machinery workshops will be required. These workshops will include both heavy and light vehicles.

The heavy machinery workshops will be designed to service production and development drills, as well as haulage equipment including loaders and trucks. While the majority of truck servicing is expected to occur on the surface, particularly during the ramp-up phase of operations, the underground workshops will provide critical support for these and other heavy machinery required for daily mining activities, especially towards the later stages of mine life.

Light vehicle workshops will be included in the base plan to do critical servicing, breakdowns and daily's. Longer period serving and major rebuilds / repairs will be conducted on the surface.

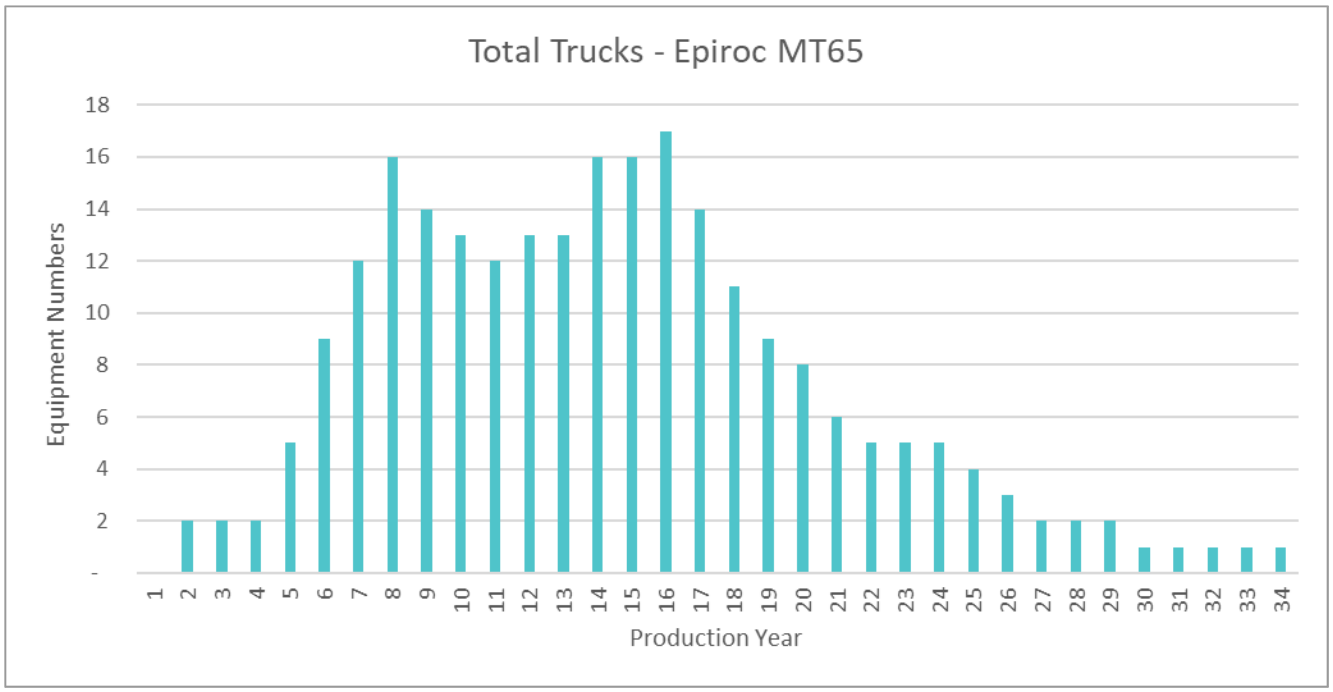
The exact sizing of these workshops has yet to be determined. However, the mine development plan has considered the need for flexibility, allowing for future revisions and expansions of these facilities as operational requirements evolve.

16.15.6 Kakula West - West equipment requirements

16.15.6.1 Mobile equipment fleet

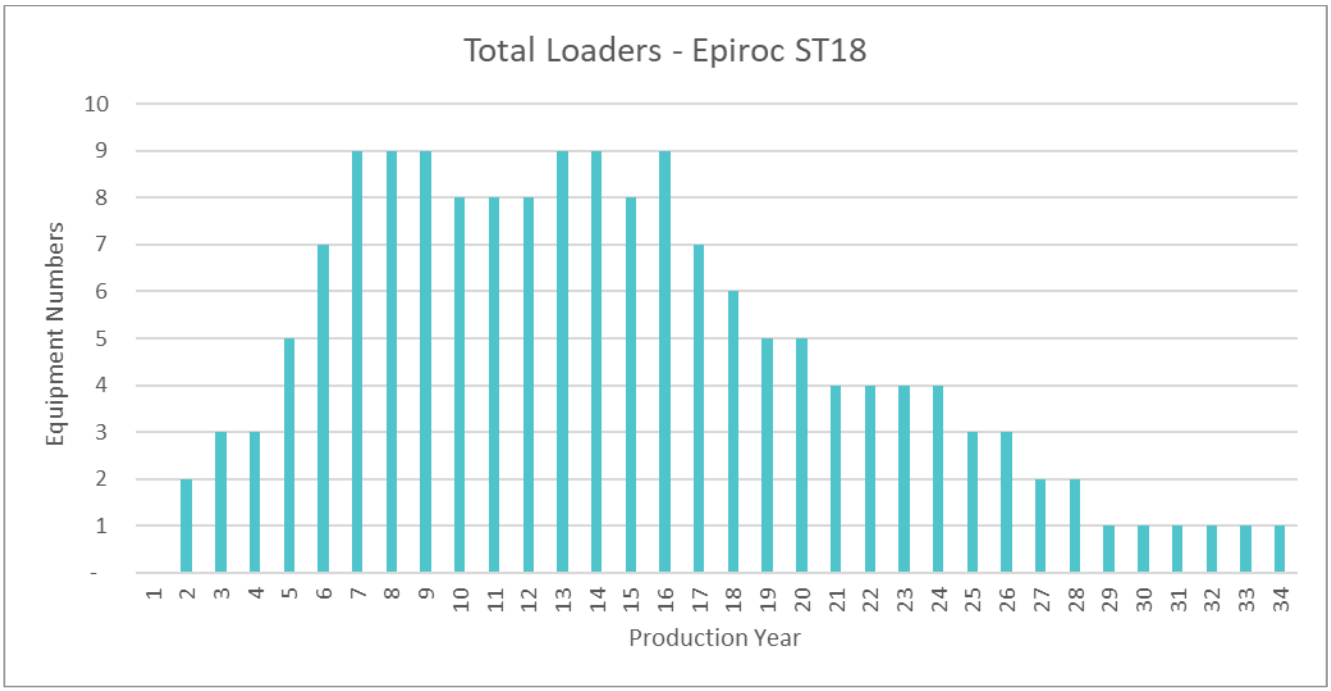
Please refer to the below figures for planned equipment requirements throughout the mine life.

Figure 16.126 Kakula West – West truck equipment schedule



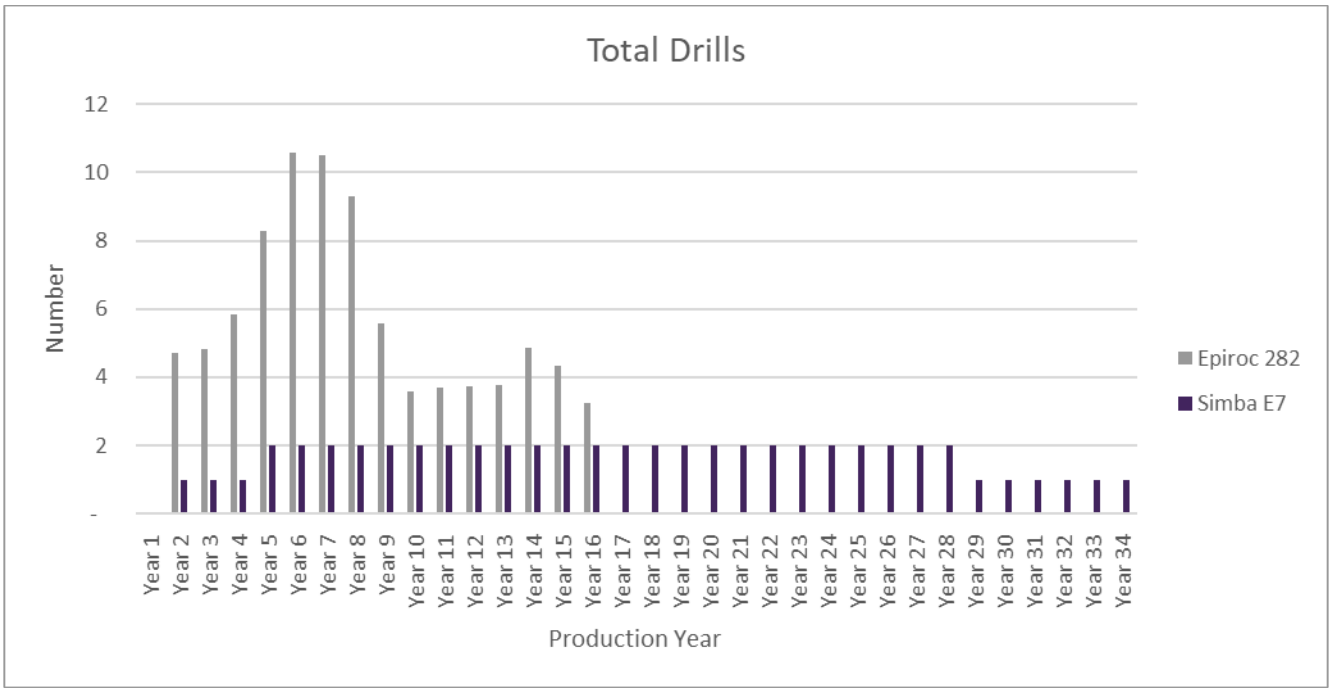
Source: AMC 2026

Figure 16.127 Kakula West – West loader equipment schedule



Source: AMC 2026

Figure 16.128 Kakula West – West drill equipment schedule



Source: AMC 2026

16.15.6.2 Fixed equipment

A conveyor and loading station will be required from Year 4 as production commences and annual lateral capital development total metres continue to increase for several years.

16.15.7 Recommendations and next steps

The following items should be considered for the next phase of a Kakula West - West study.

- Geotechnical assessment of LHS stope design parameters. The potential increase in LHS stope length and level spacing reduces lateral development and increases productivity and efficiency.
- Reducing minimum LHS stope mining width of 4 metres due to operational limitations would result in an increase in recoverable mineable ore inventory.
- The 7.5 m stoping height limitation should be reviewed in the context of additional numerical modelling and optimized pillar placement.
- Investigate placement of regional pillars according to orebody geometry.
- Further analysis of the integration of the Kakula West – West and East mining areas to establish optimal access and ore handling capabilities.

16.16 Kamoa-Kakula production profile

The combined mining profile for the deposits of KCSA are described in Table 16.71. The sequence of mining has been aligned to maintain a consistent production profile of 17Mtpa. The Mineral Reserve plan achieves steady production of 17Mtpa between 2028 and 2050, with a tapering tail of reducing production as deposits are depleted.

The mining sequence is developed to match timelines for establishment, development and production to maintain appropriate blends of ore feed to the concentrators, concentrate blend to the smelter and copper grade. Some deposits exhibit a low production “tail” during retreat recovery of regional stability pillars, and this tail will require further analysis to increase recovery rates and increase productivity.

Table 16.71 KCSA deposits combined mining production

Existing Kamoa Mine	Ore Tonnes (Mt)	Grade (%Cu)	Cu Metal (Mt)	Peak Production (Mtpa)	Steady Production (Mtpa)	Production Life (Yrs)
Kamoa 1	103.9	2.7	2.8	5.5	4.0	26
Kamoa 2 Central and East	57.0	2.6	1.5	3.3	2.1	27
Kamoa 2 West	21.0	2.5	0.5	2.8	1.8	12
Kansoko Sud	32.5	2.8	0.9	2.6	1.2	26
Kakula	51.3	3.9	2.0	8.0	6.5	8
Subtotal	266	2.9	7.8	-	-	-

Future Operations	Ore Tonnes (Mt)	Grade (%Cu)	Cu Metal (Mt)	Peak Production (Mtpa)	Steady Production (Mtpa)	Life of Production (Yrs)
Kakula West - East	52.5	2.5%	1.3	3.0	1.6	37
Kakula West - West	31.9	3.8%	1.2	2.2	1.5	30
Kamoa 3	57.6	2.4%	1.4	3.5	2.4	21
Kamoa 4	43.0	2.5%	1.1	3.9	2.5	17
Kamoa 5	8.6	2.7%	0.2	1.5	0.7	12
Kamoa 6	7.2	2.7%	0.1	1.3	0.6	9
Subtotal	201	2.7%	5.3			
Total	**466**	**2.8%**	**13.1**			

Note: Steady production for individual deposits defines the average production rate excluding build-up and ramp-down production periods.

17 Recovery methods

17.1 Introduction

The Kamoa-Kakula MRMR Update Technical Report considers a total concentrator production capacity of approximately 17 Mtpa. This is based on the modified capacity of the Kakula Phase 1 and Phase 2 concentrators of 10.5 Mtpa, together with the achieved operating capacity of approximately 6.5 Mtpa from the Kamoa Phase 3 concentrator.

Each module of the existing Kakula concentrator is undergoing several modifications and hydraulic upgrades to increase the Kakula complex capacity to 10.5 Mtpa and improve circuit recovery above 90% at the planned head grade. The Kamoa concentrator, commissioned in June 2024, was designed for 5.0 Mtpa and has consistently operated above design capacity, achieving an annual average throughput of 6.5 Mtpa while maintaining recoveries above 86% at a head grade of approximately 2.3%

This section details the process and engineering design basis of the Kamoa Concentrator Plant and the Kakula Optimization (P95) Project. The Kakula and Kamoa concentrator process designs are based on test work 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 optimization Project is aimed at increasing current recovery at the 10.5 Mtpa Kakula complex to 90% by making plant modifications described in the process design criteria. The Kakula concentrator design criteria are shown in Table 17.1.

Table 17.1 Kakula concentrator plants design criteria

Design Parameter	Unit	Design Value
Annual Surface Crushing Circuit Feed	Mtpa	10.5
Surface Crushing Circuit Availability	%	74
Surface Crushing Circuit Operating Time	hpa	6,482
Surface Crushing Circuit Feed Rate	t/h	1,562
Annual Milling Circuit Feed	Mtpa	10.5
Overall Milling Circuit Availability	%	91.3
Milling Circuit Operating Time (hours per annum - hpa)	hpa	7,998
Milling Circuit Feed Rate	t/h	1,312
Milling Module Feed Rate	t/h/module	656
ROM Cu Grade	% Cu	3.34
Final Concentrate Grade	% Cu	47.0
Mass Pull to Final Concentrate	% Mill Feed	6.4
Cu Recovery	**%**	**90.6**

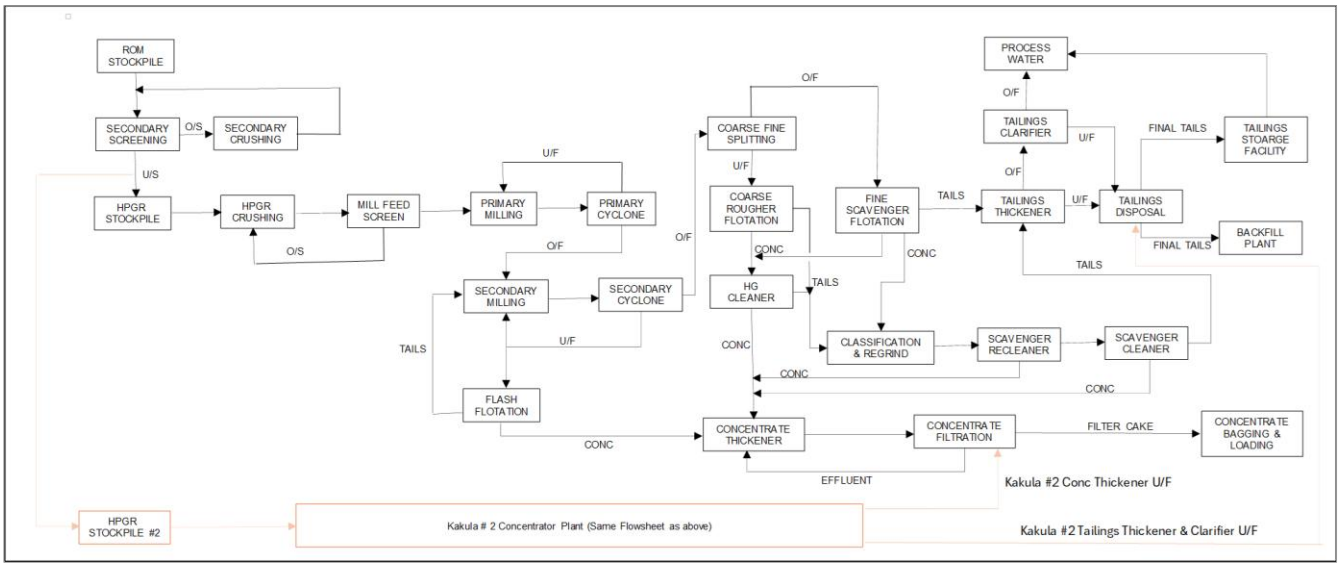
17.2.2 Plant design and process description

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 with reconfigured cleaner flotation circuits, complete with upgraded concentrate regrind circuits.
- Two identical concentrate thickening, filtration, bagging and loading systems.
- Two identical tailings thickening and tailings clarifier circuits.
- Shared tailings disposal and backfill plant feed systems.

A high-level block flow diagram of the Kakula concentrator plant is shown in Figure 17.1.

Figure 17.1 Kakula Concentrators P95 block flow diagram



Source KCSA 2025

The following modifications are being made to increase the recovery of Kakula Concentrator above 90% and guarantee a processing rate of 10,5Mtpa (5,25Mtpa per module).

17.2.2.1 Milling circuit

The milling circuit modifications are in the secondary mill and are aimed at recovering liberated copper at the earliest possible opportunity in the circuit. The main equipment changes in the milling circuit include:

- Installation of a flash flotation cell to process half of the secondary mill cyclone underflow.
- Installation of coarse-fine split cyclone cluster circuit complete with coarse rougher flotation feed pumping system and fine scavenger flotation feed pumping system.

17.2.2.2 Flotation circuit

The flotation circuit modifications are aimed at increasing recovery without sacrificing concentrate grade. The main changes in the flotation section include:

- Reconfiguring the rougher/scavenger flotation bank by:
  - Converting the second rougher flotation cell to a coarse rougher flotation cell, including installation of new tails back pressure pipe to route tails to a new regrind circuit
  - Converting the last six cells of the rougher scavenger flotation bank to fine scavenger flotation cells by installing a new feedbox on the third flotation cell.
  - Converting the first rougher flotation cell to a scavenger cleaner flotation cell for additional scavenger cleaning capacity
  - Several pump and pipe upgrades

17.2.2.3 Concentrate regrind circuit

The regrind circuit upgrade and reconfiguration is aimed at processing the coarse rougher flotation tails and fine scavenger flotation concentrate of the reconfigured flotation circuit. The regrind circuit will have two new additional 5.5MW HIGMills per module consisting of:

- Classification circuit consisting of a cyclone feed surge tank and pumping system, and a cyclone cluster.
- Common regrind mill feed tank and dedicated mill feed pumps. The pumps will be installed in a duty/standby configuration.
- Vertical tower mills (HIGMills)
- Regrind mill product tank and pumping system, with the pumps installed in a duty/standby configuration.

The regrind mills will be installed in parallel configuration.

17.2.2.4 Thickening circuit

The reconfigured flotation circuit will produce four concentrates, flash flotation concentrate, high grade cleaner concentrate, scavenger cleaner concentrate and low grade cleaner concentrate. The existing two concentrate thickening circuits, a dedicated one for each plant, have sufficient capacity to handle all the concentrate produced.

A 3 m^2 concentrate thickener feed trash removal screen, which will remove any trash/foreign objects/fibres from the final concentrate product is installed ahead of 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. The existing 2-in-1 sampling system will be used to sample the final concentrate.

A temporary concentrate storage pond is used to store any overflow concentrate produced.

The final tailings thickener circuit will be upgraded to improve the clarity of the thickener overflow. A new tailings thickener will be provided for each module. The existing tailings thickeners will be converted to clarifiers. The tailings disposal system will not be modified.

17.2.2.5 Concentrate filtration

The four Larox Filter units (PF 72/72 M60) have sufficient capacity to cater for the concentrate production.

The filtration feed tank area has three mechanically agitated filtration feed tanks (each of 500 m^3). The first and second tanks are fitted with three filter feed pump sets (in running/standby configuration) feeding to Larox Filter #1, Larox Filter #2 and Larox Filter #3. Material from the third feed tank can report to both Larox Filter #1 and Larox Filter #4. Filter #4 is fed from the third tank using two pumps in running/standby installation.

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 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 reports to the concentrate thickening circuit.

Auxiliary systems for the four filter press units include hydraulic pressing systems, filtrate and manifold wash systems, pressing air compressors and air receivers, and drying air compressors and air receivers.

17.2.2.6 Tailings disposal

Modifications in the tailings disposal circuit are focused 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 are being upgraded to achieve the higher flowrate requirement from the additional pumping systems.

17.2.2.7 Services and reagents

The instrument air system is being upgraded by providing an additional compressor, instrument air receiver for the new regrind circuit and an air receiver for the new tailings thickener area. The flocculant and coagulant distribution pumps are being upgraded. A new reagent (depressant/sodium silicate) makeup and dosing system for fine scavenger flotation will be installed. Provision is made for installation of frother and collector dosing pumps to cater for new dosing points introduced by the flotation circuit reconfiguration. Additional hosing water tank and pumps systems are being installed in the new tailings thickener area.

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 test work findings and assessments as presented in Section 13, various trade off studies, and relevant design information. The design of the process plant was undertaken by DRA Projects (PTY) Ltd (DRA Projects).

The initial design philosophy adopted a modular approach, providing flexibility to expand from a single 5 Mtpa module to two modules with a combined capacity of 10 Mtpa. The first 5 Mtpa module was

commissioned in 2024. The Kamoa concentrator is currently operating approximately 40% above its design capacity, resulting in an effective nameplate capacity of approximately 6.5 Mtpa for the current circuit.

17.3.2 Kamoa concentrator basis of design

The Kamoa concentrator includes all ore processing requirements from the run-of-mine stockpile through to final concentrate dispatch and final tailings disposal. The original Kamoa concentrator design criteria and the Kamoa concentrator upgrade design criteria are shown in Table 17.2. The overall plant recovery is negatively impacted by the increased supergene content in the 2026 PFS mining production schedule.

Table 17.2 Kamoa concentrator process design criteria

Design Parameter	Unit	Upgrade Design Value
Annual Surface Crushing Circuit Feed	Mtpa	6.5
Surface Crushing Circuit Availability	%	65
Surface Crushing Circuit Operating Time	hpa	5,734
Surface Crushing Circuit Feed Rate	t/h	1,134
Annual Milling Circuit Feed	Mtpa	6.5
Overall Milling Circuit Availability	%	88
Milling Circuit Operating Time	hpa	7,709
Milling Circuit Feed Rate	t/h	850
Milling Module Feed Rate	t/h/module	850
ROM Cu Grade	% Cu	2.46
Final Concentrate Grade	% Cu	35.1
Mass Pull to Final Concentrate	% Mill Feed	5.83
Cu Recovery	**%**	**83.0**

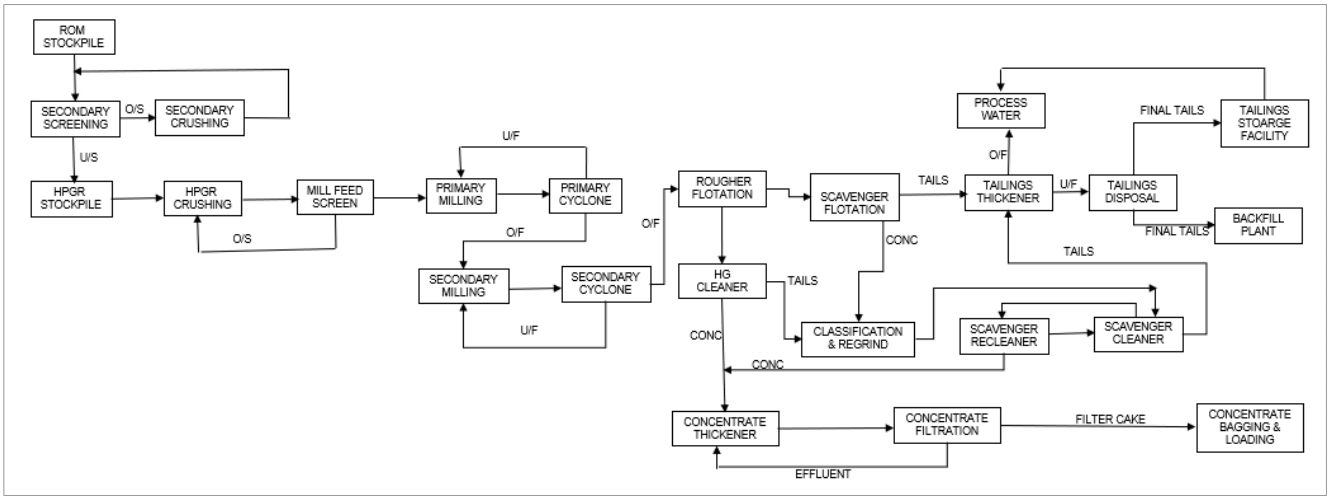
17.3.3 Plant design and process description

The concentrator consists of the following:

- Crushing and screening module.
- High-pressure grinding rolls (HPGR) crushing circuit in series with a primary ball mill circuit.
- Secondary ball mill circuit.
- Flotation circuit complete with concentrate regrind circuit.
- Concentrate thickening circuit.
- Concentrate filtration circuit.
- Tailings thickener with tailings thickener overflow clarifier.
- Final tailings disposal system and backfill plant feed system.
- Utility systems for compressed instruments air, water supply and reagent makeup and distribution systems.

Figure 17.2 is a high-level block flow diagram for the Kamoa concentrator.

Figure 17.2 Kamoa concentrator block flow diagram



Source KCSA 2025

The following sections describe the main components of the Kamoa concentrator.

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. Dust suppression is provided in this area.

17.3.3.2 Crushing and screening

The material from the stockpile combines with the primary crusher product on the primary screening feed conveyor before discharging into 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 the HPGR feed stockpile via the primary screen undersize transfer conveyors.

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 two continuously operating cone crushers (Model: CS660). Each primary cone crusher is installed with a 315 kW motor to achieve a size reduction from F80 195 mm to P80 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.

17.3.3.3 HPGR stockpiling

The primary screening undersize product from the screen is conveyed by dedicated conveyors to the HPGR feed stockpile. The HPGR feed stockpile is designed to store a live capacity of 7,500 t. The material is

extracted from the stockpile at a controlled rate via two variable speed belt feeders, which discharge the material onto dedicated HPGR feed bin transfer conveyors.

Provision is made for a dust control system in the area.

17.3.3.4 HPGR crushing

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 identifies 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 utilized 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 circuit consists of a 7,0m Ø x 10.37mEGL, 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 cyclones.

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, P80 152 µm, reports to the secondary mill discharge sump as feed.

70 mm high chrome steel balls are loaded to the primary mill feed hopper via the primary mill feed screen underpan using bags.

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. A mill relining machine is provided.

17.3.3.6 Secondary milling

The secondary milling circuit consists of a7.0mØ × 10.37m EGL overflow discharge ball mill (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 pumped 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 (P80 57 µm) reports to the mechanical agitated, 500 m^3 rougher flotation feed surge tank via a two-stage metal accounting sampling installation and trash removal system. Reagents are dosed into the rougher feed surge tank at controlled via dedicated dosing funnels, using dosing pumps. Milling circuit product is pumped to the rougher flotation feed box using variable speed, duty / standby, pumping systems.

A magnet and sputnik arrangement is used to load 30 mm high chrome steel balls to the secondary mill feed hopper. Spillage collection and pumping systems are installed at the rougher flotation feed tank area.

17.3.3.7 Rougher /scavenger flotation

Milling circuit product is pumped to the head of the rougher flotation circuit at a controlled rate and density, where frother is dosed. 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.

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 mixed collector (a xanthate and a dithiophosphate) 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.

17.3.3.8 High-grade cleaner flotation

The high-grade rougher concentrate gravitates to the high-grade cleaner feed sump, from where it is pumped to the high-grade cleaner flotation circuit via a trash removing screen. The high-grade cleaner flotation circuit, consisting of a single low entrainment Jameson flotation cell produces the final high-grade concentrate product. Filtered water is provided as froth washing water to the Jameson 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 analyzer for monitoring of the high-grade concentrate grade.

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 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 F80 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 P80 15 µm, which bypasses the regrind mills directly to the 60 m^3 regrind mill product sump.

The cyclone underflow product (P80 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 15-20 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 circuit 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.

17.3.3.10 Scavenger cleaner flotation

The scavenger cleaner flotation circuit comprises of a single bank of nine160 m^3 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 m^3 low-grade concentrate sump from where it is pumped to the scavenger recleaner circuit using a variable speed, duty / standby pump system.

The medium-grade concentrate gravitates to a 7 m^3 sump from where it is pumped to the concentrate thickener circuit using a fixed speed, duty / standby pump system.

The scavenger cleaner tailings gravitate to the 45 m^3 scavenger cleaner tailings sump via a two-stage sampling system from where it is pumped to the tailings thickener, to combine with the scavenger tailings product.

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 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 analyzer.

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 flotation tailings consisting of scavenger tailings and the scavenger cleaner tailings are pumped to feed box of a 47 m diameter, high-rate thickener unit. Flocculant is added to the tailings in the feed box. The tailings then gravitate to the thickener feedwell. The thickener design allows for auto dilution.

The tailings are thickened to an underflow product targeting 55 % solids (w/w), before being pumped to the backfill feed surge tank, via a two-stage sampling system, using variable speed, duty / standby pumps. Tailings thickener overflow products gravitate to the tailings clarifier, 50 m diameter unit. The clarifier overflow gravitates to a 3,000 m^3 process water tank for reuse as process water. The underflow is pumped to the backfill feed surge tank via the two-stage sampling system, using variable speed, duty / standby pumps.

Spillage produced in the tailings thickening area gravitate to spillage collection sumps from where it is pumped to the thickener feed box.

17.3.3.13 Backfill feed system and final tailings disposal

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, if required to obtain densities suited for long distance pumping 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 circuit consists of consists a 24 m diameter, high-rate thickener unit. High-grade cleaner concentrate, medium grade concentrate and recleaner concentrate are pumped to a common two-in-one sampling system. The combined sampled concentrate gravitates to a trash removal linear screen. Linear screen undersize gravitates to the thickener feed box and oversize trash gravitates to a trash bin for further handling. Flocculant at a controlled rate is added to the filter feed box.

The concentrate thickener unit is fitted with an automatic internal dilution system for 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 centrifugal 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 tanks feed box. The concentrate can gravitate to either of two 520 m^3 mechanically agitated filtration feed tanks, from where it is pumped at a controlled rate to one of three horizontal plate pressure filters using a variable speed, duty / standby pump set.

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.

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 horizontal plate pressure filters with a total filtration area of 156 m^2, in two duty and one standby configuration.

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 reports 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 has an overhead travelling crane for use during maintenance.

17.3.3.17 Air services

Flotation blower air

Low-pressure blower air to the forced air tank flotation cells is supplied by three fixed speed multistage centrifugal air blowers, of which two are duty and one standby. Each blower is equipped with a dedicated suction filter, and suction silencers. In addition to silencers fitted on each suction, each blower is further equipped with dedicated delivery line silencer units.

Compressed air

Compressed instruments air is supplied from two air compressors complete with air dryers supplied as vendor packages, dedicated to the concentrator plant. The compressors are installed in a duty/standby configuration.

The compressed air is generated at 1,300 kPa, passed through duty / standby air filtering and drying systems before being stored in a 10 m^3 instrument air receiver. An additional 10 m^3 instrument air receiver is placed in the milling area. 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.

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

Process water, made up of any excess concentrate thickener effluent, TSF return water, and tailings thickener overflow products is stored in a 3,000 m^3 process water tank. The process water is required for

dilution. It is distributed to various sections of the concentrator by dedicated duty / standby process water pumps.

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

Filtered water is supplied from a sand filter plant. The filtered water is pumped to a 1,500 m^3 storage tank from where it is distributed across the concentrator plant for use as gland seal water, dust suppression, mill cooling water and Jameson cell spray water.

The gland seal water pump circuit is designed as a separate system due to higher-pressure requirements and includes dedicated pressure-controlled gland seal water ring mains to the milling-flotation areas, fed from duty / standby gland seal water pumps. Other raw water is distributed to the required points at a controlled pressure using the duty / standby concentrator raw water pumps.

Potable water

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.

A fire water system, consisting of a 1,000 m^3 storage tank and distribution pump system is provided for.

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 in 850 kg bags and stored in the reagent store. The bags are moved from the reagent store to the SIBX make-up area when required.

During batch make-up, a bag is manually hoisted and the contents discharged into the 25 m^3 mechanically agitated collector mixing tank. The powder collector is diluted with potable water to 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 storage tank. From here it is distributed to the 10 m^3 day tank using a duty / standby peristaltic pump system. Dedicated dosing pumps will pump the required volumetric flow of collector solution from the day tanks to each dosing point.

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 using a vertical spindle pump. Provision is made for safety showers in the area, as well as flash back arrestors on each storage tank.

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 Intermediate Bulk Containers (IBCs).

The design includes for duty / standby promoter transfer pumps to transfer the liquid from the 1 t IBCs 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.

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.

The IBCs are located within the flotation area bunds and no additional spillage handling system is required.

17.3.3.22 Flocculant make-up and dosing

BASF Magnafloc 10 flocculant is used at the concentrate and 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.

Potable water is used in both the mixing / dosing tanks for mixing and diluting to the transfer strength of 0.5% (w/v). The mixed flocculant is allowed to hydrate for about 90 minutes before it is pumped to the respective thickening circuits using duty / standby pump systems. The flocculant is further diluted to 0.05% (w/v) with filtered water in inline mixers before each the dosing point.

Spillage produced in the flocculant make-up and dosing area is collected in a dedicated spillage sump and pumped to the final tailings disposal sump via a vertical spindle pump.

17.3.3.23 Coagulant make-up and dosing

Coagulant is added to the tailings thickener 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 transfer the dry coagulant into either one of the two 100 m^3 mixing / dosing tanks, via vendor supplied wetting systems.

Potable water is used in both mixing / dosing tanks for mixing and diluting to the transfer strength of 0.5% (w/v). The mixed coagulant is allowed to hydrate for about 90 minutes before it is pumped to the tailings thickener and tailings thickener clarifier thickening circuits using duty / standby pump systems. The coagulant is further diluted to 0.05% (w/v) with filtered water in the inline mixers before each dosing point.

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.

17.3.3.24 Concentrator services requirements

Table 17.3 lists the water, consumables and power requirements for the concentrator.

Table 17.3 Concentrator Services

Item	Description	Consumption per tonne of Plant Feed	Annual Requirement
Power	Electricity	45.0 kWh/t	292.5GWh
Water	Raw make-up	0.20 m^3/t	1,300 ML
Reagents	Frother	180 g/t	1,170 t
	Collector	180 g/t	1,170 t
	Promotor	18 g/t	117 t
	Flocculant	75 g/t	352 t
	Coagulant	18g/t	168 t
	Lime	680 g/t	4,420t
Consumables	Grinding media (70 mm steel balls)	0.350 kg/t	2,275 t
	Grinding media (30 mm steel balls)	0.370 kg/t	2,405 t
	Grinding media (3 mm Ceramic)	**15g/t**	**97.5 t**

17.4 Kamoa-Kakula copper smelter

17.4.1 Smelter development timeline

The concept of using direct to blister flash (DBF) smelting technology for the Kamoa was mooted as far back as 2013 on the basis of a PEA produced in 2012. The description provided in the 2013 NI 43-101 report on the project is very similar to what is on the ground today. The 43-101 report has been updated six times since 2013 with unwavering commitment to the DBF concept and this is surely testament to the predictable nature of the ore and its mineralogy right from the start. At the end of 2021 Kamoa copper awarded China Nerin Engineering a basic engineering contract for a 500,000 tpa copper smelter utilizing the DBF technology licensed by Metso. Metso supplied key parts of this basic engineering and were to be preferred suppliers of certain core technology equipment. In 2022 Metso: Outotec conducted continuous flash smelting test work using their pilot plant in Pori, Finland. The test work was intended to verify the design parameters for the furnace and to identify the operational window. The final EPCM contract was awarded to Nerin in 2022 with commissioning expected at the end of 2024. Construction and cold commissioning of the smelter was completed in May 2025, with hot commissioning starting only in December due in part to feed scarcity and power stability concerns. Power stability issues were resolved through the installation. of a 60MW uninterruptible power supply. First anode was produced in December 2025. To date, the smelter ramp-up has progressed smoothly with current production at nearly 70% of nameplate.

17.4.2 Kamoa-Kakula Smelter Process design

17.4.2.1 Smelter design criteria

The smelter is designed to produce about 500,000 tpa of copper anode from about 1.2 million tons of combined Kakula / Kamoa concentrates. The major items in the process design criteria are included in the table below.

Table 17.4 Smelter Design Criteria

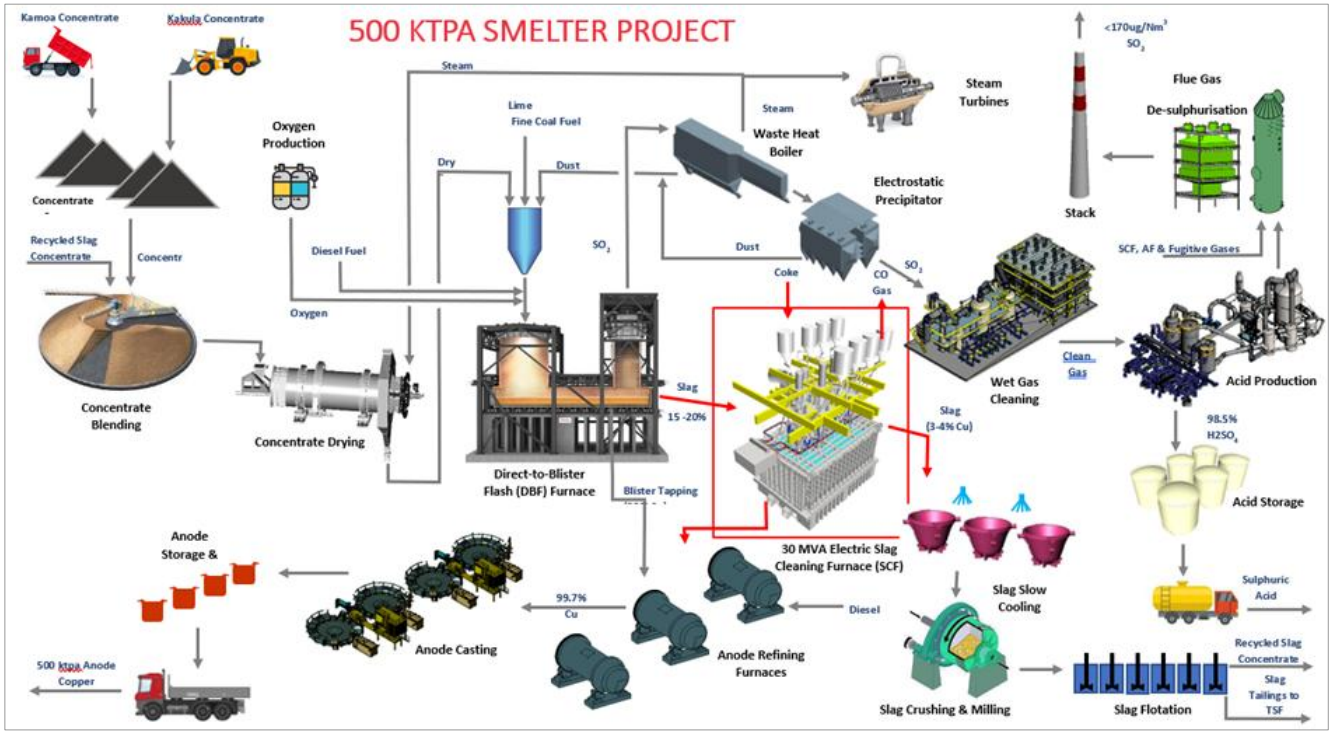
Item	Units	Overall	
Processed Material Streams		Kamoa	Kakula
Blended Concentrate Operating Range	%	20% to 70% of total	80% to 30% of total
Concentrate Composition Range	% Cu	36.7	39.6 to 58.5
	% Fe	17.9	3.0 to 5.0
	% S	25.0	10.7 to 16.2
Major Sulphide Mineral Content	% (>5%)	Chalcopyrite 31%, Bornite 22%, Chalcocite 13%, Pyrite 8%	Chalcocite 61%, Bornite 6%
Concentrate Physical Characteristics		Bulk Density 2.21 t/m3, 12% H2O, 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+O2<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	
Sulphuric Acid Product Quality	%	H2SO4>98.5	
Equipment Operating Conditions			
Smelter and Acid Plant Operating Time	Hrs. /Yr	7400 (84.5%)	
Gaseous emission standards	mg/Nm3	SO2 200, NO x 300, dust 5	
Item	Units	Overall	
Sulphuric acid production	ktpa	783	
Concentrate Filter Cake Storage Capacity	Days	14	
DBF Operating Temperature	Degrees Celsius	Blister Copper 1,250, DBF Slag 1,300, Off gas<1,370	
SCF Operating Temperature	Degrees Celsius	Blister Copper 1,250, DBF Slag 1,350, Off gas 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 CO2 Ratio is 1.02 During Settling, 5.38 During Reduction	
Tail Gas from Acid Plant and Desulphurization	**mg/Nm3**	**SO2<170**	

Source: Orewin, 2023.

17.4.2.2 Smelter process flow diagram

The smelter process is shown in the pictorial flow diagram below.

Figure 17.3 Smelter Process Flow



Source: KCSA, 2025.

17.4.3 Raw material handling (Flash furnace)

Concentrates from Kakula and Kamoa together with burnt lime and coal are received at the smelter.

17.4.3.1 Concentrate handling

Concentrate from the adjacent Kakula concentrator is loaded with a front-end loader into a loading hopper above a transfer conveyor. Concentrate from the remote Kamoa concentrator is loaded from trucks into a loading hopper above the same transfer conveyor.

Concentrates are transferred separately to two separate stockpiles in a large storage shed. Each stockpile contains up to 15000 tons or one week's supply.

Stockpile reclaimers withdraw from the two stockpiles at the required ratio onto the blended feed conveyor.

A separate loading system using a front-end loader, loading hopper and belt feeder deposits concentrate recovered from slag onto the same conveyor. There is storage capacity for 5000 tons or about 14 days of slag concentrate.

The blended feed conveyor discharges to a circular covered blending stockpile.

A circular reclaimer withdraws blended concentrate onto a conveyor that feeds a vibrating trash screen. Trash reports to a waste bin and clean concentrate containing about 12% moisture reports to the steam drying plant.

17.4.3.2 Coal handling

It was envisaged that pulverized coal would be required as an addition fuel source for the DBF when very low energy concentrates were to be processed. A pulverized coal preparation plant having a capacity of about 20 tons per day is therefore provided.

In more recent times the use of pyrite to replace coal as an energy source has been agreed upon, subject of course to availability. The advantage of using pyrite is that is also a source of additional sulphur units to supplement acid production.

17.4.3.3 Lime handling

Granulated burnt lime is received in bulk tankers onsite. Tankers are offloaded pneumatically to two lime silos each having a capacity of 3 to 4 days usage.

From the silos lime can be transferred pneumatically to a day bin close to the flash furnace.

Another pneumatic transfer system conveys lime as required to the lime powder bin on top of the flash furnace.

17.4.4 Raw material handling (Slag cleaning furnace) (SCF)

Apart from the molten slag feed from the flash furnace the SCF reprocesses slag from the anode furnaces and uses coke to effect reduction of the copper in the slag to metal.

17.4.4.1 Coke handling

Coke is stored undercover in a storage yard. There is capacity to hold 2,500 tons of coke sufficient for one month's usage in the SCF. Coke is transferred to six dedicated SCF feed bins located on top of the furnace using a front-end loader via a loading hopper, weigh feeder, transfer conveyor and tripper conveyors.

17.4.4.2 Anode refining slag handling

Cooled anode refining slag is crushed to about 60mm using a jaw crusher and then transferred to a covered storage yard. Using the same transfer system as used for the coke, AF slag is transferred to two dedicated AF slag feed bins located on top of the furnace.

17.4.5 Concentrate steam drying

A steam tube dryer utilizing steam generate from the flash furnace waste heat boiler is employed to dry concentrates received to a 'bone dry' state. Air carrying evaporated moisture is treated through a bag filter while dry concentrate is screened at 2mm prior to being transferred pneumatically to the insulated concentrate feed bin. +2mm-20mm lumps are comminuted in a hammer mill and join the -2mm fraction. +2mm lumps are recycled to the concentrate shed.

Dryer capacity is 200 tph (wet basis) of concentrates containing up to 12% moisture. There is no design margin available to treat Kamoa : Kakula ratios exceeding 70:30.

17.4.6 Flash furnace feed system

As noted in the sections above DBF feed materials are stored in dedicated feed bins on top of the flash furnace. An additional bin for flue dusts from the waste heat boiler, ESP, SCF and the anode furnaces is also provided.

Loss of weight feed systems are used to blend the flash furnace feed materials in the appropriate proportions. The blended mix is fed to the DBF concentrate burner using an air slide.

17.4.7 Flash furnace (DBF)

The concentrate blend, lime, pulverized coal and recirculated dusts are fed to the concentrate burner. The feed charge and oxygen enriched air enter the vertical reaction shaft of the flash furnace where the temperature is controlled by the degree of oxygen enrichment and the strategic placement of diesel / oxygen burners in the reaction shaft and the settling area freeboard.

Oxygen is supplied from a VPSA oxygen plant later described.

The sulphide minerals are rapidly converted to a mixture of elemental copper and metal oxides according to the following equations:

- Cu_2S (chalcocite) + O_2 --> 2Cu + SO_2
- Cu5FeS_4 (bornite) + 4.5O_2 --> 5Cu + FeO + 4SO_2
- $CuFeS_2$ (chalcopyrite) + 2.5O_2 --> Cu + FeO + 2SO_2
- 2Cu + 0.5O_2 --> Cu_2O
- 2FeO +SiO_2 --> 2FeO.SiO_2 (fayalite)
- 3FeO + 0.5O_2 --> Fe_3O_4 (magnetite)
- SiO_2 + CaO --> CaO.SiO_2

The molten product mixture together with the off-gas report to the horizontal charge settling area where blister copper separates from the slag components. Dust laden off gas escapes the furnace through the vertical gas offtake shaft. Per the test work described in section 17.4.2 above it is important to control the copper content in the slag in the range 16 to 22%Cu so that the sulphur content of the blister is below 0.2% and there is minimal risk of foaming. The settling area of the flash furnace is equipped with tap holes for the molten blister copper and the slag. Molten blister copper is tapped periodically and routed to the anode furnaces via launders. Copper rich slag is tapped periodically and directed via launders to the slag cleaning furnace.

Extensive use is made of copper cooling elements in the furnace refractory systems to deal with the aggressive nature of the slag and the blister copper at the operating temperature of around 1300 deg C.

Depending on the Kamoa: Kakula ratio the blister copper fall in the DBF varies between 70% and 85% of the total blister production, the remaining fall occurs in the SCF

The exothermic nature of the oxidation reactions means that this furnace is suited ideally for sulphur deficient or low energy concentrates. It is important that the proportion of high energy chalcopyritic concentrates is limited in the feed blend to prevent overheating, excessive slag generation, potential for slag foaming and damage to the furnace lining systems.

Indicatively, this situation can prevail when the Kamoa: Kakula concentrate ratio exceeds 70:30. Under such conditions the FSF feed rate will have to be reduced and excess concentrate toll treated elsewhere. Quite clearly the maximum Kakula: Kamoa concentrate ratio that can reasonably be treated will be determined through plant operation.

Late in the life-of-mine, when the Kakula resource is exhausted, there is a plan in place to introduce a double flash process (flash furnace / flash converter) more suited to energy rich chalcopyritic concentrates.

17.4.8 Slag cleaning furnace (SCF)

Oxidized slag containing about 20% copper from the flash furnace is subject to reduction using metallurgical coke in a unique 6 electrode rectangular arc furnace. The purpose is to reduce oxidized copper back to copper metal. The chemistry is as follows

$$Cu_2O + C \rightarrow 2Cu + CO$$

$$FeO + C \rightarrow Fe + CO$$

These reactions are endothermic, so additional energy has to be supplied through electric power. It is important to minimize the extent of the second two reactions so that the purity of the copper is maintained. This is achieved by keeping copper levels in the discharge slag at about 3%. The slag cleaning furnace is equipped with separate tap holes for slag and metal. Copper is periodically tapped via launders to the anode furnaces. Slag is periodically tapped into cast iron ladles which are transported to a slow cooling yard. The purpose of slow cooling is to allow proper phase separation of residual sulphides and copper metal in a crystalline slag. As will later be seen the subsequent recovery of copper from the slag is thereby enhanced. Carbon monoxide rich off gas reports to a combustion chamber described later in the gas cleaning section.

A critical aspect of operating the SCF is proper control of the operating temperature and the slag chemistry so that the integrity of the refractory linings can be maintained. With this in mind and understanding that the SCF operates in batch mode with distinct operating periods:

- 2hrs for slag reduction
- 2 hrs. for slag settling
- 2hrs for slag tapping

The manner in which power input is applied is important. The furnace has 6 electrodes arranged in two three electrode deltas that have separate power supplies. This enables more even power distribution across the width of the furnace and an ability to distribute power differently between the front and the back of the furnace. During the reduction cycle more power is applied at the front of the furnace to satisfy the needs of the reduction process with less power at the back for the settling process. Once reduction is complete the power at the front can be reduced for further settling.

Extensive use of copper cooling elements in the refractory lining systems of this furnace is made.

Soderberg type graphite electrodes with associated systems for paste addition, casing addition and electrode baking are provided. Electrode diameter is 1200mm with each delta supplied by a 15MVA 3 phase transformer. Expected power input during reduction is 22MW (peak) 14MW (average) reducing to less than 5MW during settling and tapping.

The SCF design is based on a maximum slag rate of 108 tph that results from the 70:30 Kamoa: Kakula concentrate ratio. The SCF has more than sufficient power and hearth area to ensure that good copper recoveries are achieved even at higher Kamoa: Kakula ratios.

17.4.9 Anode furnaces and anode casting wheels

Blister copper from the flash furnace and the slag cleaning furnace reports via launders to three 660-ton anode refining furnaces. In these anode furnaces residual sulphur and iron are blown down with oxygen / air. Additional heat requirements are provided using diesel injected with the oxygen / air. Small amounts of raw diesel are injected at the end of the cycle to ensure complete oxygen removal. Slag formed during oxidation will be decanted cooled and recycled as described above. Once the refining is complete clean copper is decanted through a launder system onto one of two twin anode casting machines each having a capacity of 110 tph. Cast anodes are cooled in a bosh tank and then transferred to an anode storage yard having the capacity to hold up to 45 days production (60,000 tons). Typically, the anodes are 99.7% copper and are dispatched by truck to market. Gases from the anode furnaces are directed to gas cleaning as later described.

Early operation of the anode furnaces has shown that there is a problem with the launder from the SCF to anode furnace three. The launder blocks at the entrance point to the anode furnace because it is a side entry as opposed to top entries on the other two furnaces that are fed from the DBF. Additional burners at the mouth of anode furnace three will be required.

Also, there is an issue with the standby ladle arrangement to transfer blister from the SCF to anode furnace 1 or 2 when no 3 is offline for maintenance. The cast iron ladles used loose to much heat and the blister freezes quickly leading to unmanageable ladle skulls. Two solutions have been suggested:

- Install a fourth anode furnace and ancillaries that would act as a standby for anode furnace three
- Obtain transfer ladles that have insulating refractory linings. To prevent freezing ladle reheat burners should also be provided. In addition, a second ladle crane will be needed

17.4.10 Off gas handling

There are four distinct off gas handling systems:

- DBF off gas a high-volume stream at high temperature containing large amounts of entrained dust and having a high sulphur dioxide content.
- SCF off gas a variable volume, variable composition stream containing some dust, significant quantities of carbon monoxide and small amounts of sulphur dioxide.
- AF off gas a variable volume, variable composition stream containing small amounts of dust and small amounts of carbon monoxide and sulphur dioxide.
- Fugitive gas collection from taphole fume extraction hoods, SCF furnace slag inlets and operational ladles. A high-volume low temperature stream containing small amounts of dust and small amounts of sulphur dioxide.

After cleaning the DBF off gas reports to a sulphuric acid plant later described.

The SCF, AF and Fugitive gases after cleaning report to a multipurpose wet scrubber plant or desulphurization plant. Two additional streams also report to this section:

- Tail gas from the sulphuric acid plant
- Off gas from the coal pulverization plant

Desulphurization is described in section 17.4.12 below.

17.4.10.1 DBF off gas handling

Dust laden DBF off gas at a temperature of 1350 deg C reports to a waste heat boiler (WHB) where it is cooled to 350 deg C. Saturated steam at 60 bar and 275 deg C is routed to a HP steam turbine later described. The WHB is designed with radiative and convective sections in such a way that accretions due to sulphation are minimized. Drag conveyors and rotary valves discharge settled dust from the bottom of the WHB to the dust handling section.

Cooled gas then reports to two hot electrostatic precipitators (ESPs). The ESPs achieve a dust removal efficiency of better than 99.5%. Dust loading is reduced from 350 g/Nm3 to 1.5 g/Nm3. The DBF exhaust fans then direct the gas to wet scrubbing. For hot gas handling there is a 45% allowance made for leakage air which is substantial, on top of this there is a design allowance of 15%.

In the wet gas scrubbing section, the gas is cleaned to meet the requirements for sulphuric acid production later described.

A fairly standard flowsheet with four stages in series is employed.

- Primary dynawave scrubber to remove residual dust and to cool the gas evaporatively to about 60 deg C. Some SO_2 adsorption into the scrub liquor occurs. It is restriped from the scrub liquor with air. All contained SO_3 is adsorbed.
- Packed tower gas cooler to reduce the water content in the gas to less than 10% v/v so that acid produced is not diluted below 98.5% in the downstream acid plant.
- Secondary dynawave scrubber for solids polishing.
- Two wet electrostatic precipitators to remove any remaining solids and entrained scrub liquors.

Solids produced in the scrubbing stage are recovered from the scrub liquor by filtration. This filter cake likely contains elements such as lead, zinc, selenium and arsenic and needs to be disposed to a safe site or sold.

A weak acid stream containing about 3 tph of contained sulphuric acid must be neutralized with lime or sold to one of the neighboring mines for oxide leaching. Gypsum produced from the neutralization can be co-disposed with tailings.

The design of the wet gas scrubbing section is conservative given the allowances made for leakage air and design factor in the upstream hot gas handling

17.4.10.2 SCF off gas handling

During the reduction period off gas from the SCF containing carbon monoxide is routed to an incinerator where the CO is combusted and then cooled by dilution with ambient air to 800 deg C. The combusted gas then passes to an evaporative cooler where water sprays reduce the temperature to about 200 deg C.

During the settling and tapping periods the gas flow reduces dramatically as does the CO content. The gas passes to the incinerator where any residual CO is burnt and ambient dilution air cools the gas to about 150 deg C. The gases pass through the evaporative cooler without any need for further evaporative cooling.

During all periods the gas from the evaporative cooler reports to dedusting cyclones where most of the entrained dust is removed. Dust collected from the evaporative cooler and the cyclones is transferred to the dust handling section using dust collection bins.

The SCF exhaust fans forward the gas to the multipurpose wet scrubbing section later described.

Based on the maximum continuous flows during reduction there appears to be a healthy design margin of 54%. It is unclear though what the peak flows might be during reduction as this is dependent on the kinetics of copper oxide reduction with coke.

17.4.10.3 AF off gas handling

During all the operational periods of the anode furnaces (oxidation, reduction and holding) off gas from the three anode furnaces at about 400 deg C will pass through air to gas plate heat exchangers where the temperature will be reduced to 180° C. Each anode furnace has an extraction fan that transfers the gas to the multipurpose wet scrubbing section.

The anode furnace off gas flows are independent of the Kamoa: Kakula ratio and again there appears to be a healthy design margin of 40%.

17.4.10.4 Fugitive off gas handling

Fume collection hoods are provided over all the tapholes on the DBF and the SCF. There are additional fume hoods on the slag inlets to the SCF and on the SCF slag ladles. The gas from all these points is passed through a baghouse and transferred to the multipurpose wet scrubbing system using fume hood extraction fans. Given the highly variable nature of this gas flow the fans are fitted with variable speed drives that maintain a constant draught.

The fugitive off gas flows are independent of the Kamoa: Kakula ratio and again there appears to be a healthy design margin of 30%.

17.4.10.5 Desulphurization and effluent treatment

As noted above there are five streams to be treated through this facility repeated here for clarity:

- SCF off gas after combustion, cooling and primary dust removal.
- AF off gas after cooling.
- Fugitive off gases after primary dust removal.
- Tail gas from the sulphuric acid plant.
- Bleed gas from the coal pulverization plant.

Detailed calculations of the required volumes for each stream were determined by Nerin with healthy margins allowed. The design flowrate for the facility is 621,000Nm3/hr with a nominal flowrate of 487,000Nm3/hr.

Since the first two streams still have significant dust loadings they are pre scrubbed in a reverse jet wet scrubber. Process water is used as the scrubbing medium. A bleed stream is thickened and the solids are filtered. This filter cake likely contains elements such as lead, zinc, selenium and arsenic and is recycled to the smelter.

Dust free gas is forwarded to gas mixing chamber where it is combined with the three remaining streams.

From the mixing chamber a fan forwards the gas to an open counter current limestone slurry scrubbing tower. The limestone reacts with sulphur dioxide contained in the gases to form calcium sulphite. SO2 removal is about 92%. In the bottom of the scrubbing tower the sulphite is oxidized to sulphate using sparged air and vigorous agitation. The oxidized slurry is circulated over the tower and a bleed stream is withdrawn for downstream treatment. Clean gas passes through a wet ESP to remove any entrained solution before finally venting to atmosphere. In principle the vented gas should contain less than 100 ppm SO2 and less than 10mg/Nm3 particulates. These emission levels are well below environmentally acceptable norms.

The bleed stream is treated through a set of dewatering cyclones and a belt filter to remove the gypsum formed from the reaction of limestone with sulphur dioxide. The gypsum cake containing 15% moisture can be repulped and co disposed with flotation tailings. Recovered process water is recirculated to the scrubbing circuits.

17.4.11 Dust handling

Dust is collected from the following sources:

- Waste heat boiler. There are two dust types. Coarse dust collected from the radiative section is directed to a set of crushers before being transferred to the WHB dust bin. Fine dust from the convective section is transferred directly to the WHB dust bin.
- Dust from the hot ESPs is transferred to an ESP dust bin.
- Dust buckets are used to transfer small amounts of SCF and AF dusts to the SCF and AF dust bin.

Each of the dust bins is fitted with a pneumatic transfer system that uses compressed air to convey the dusts to a target bin located above the flash furnace. From the target bin a rotary valve directs the dust to the flash furnace dust feed bin.

Bag filters are installed to collect fugitive dusts from the various bins in the system

Fumes from the fugitive gas collection system are collected in a separate baghouse and are recycled to the concentrate yard using bulk bags.

17.4.12 Sulphuric acid plant

A modified double contact double adsorption (DCDA) sulphuric acid plant is employed to convert sulphur dioxide to sulphuric acid at an annual rate of 783,000 tons (100% H2SO4 basis). Since the gas strength after gas cleaning is in all cases higher than the maximum 12% (v/v SO2) for conventional DCDA an addition partial pre-conversion stage is employed to allow gas at 16% (v/v SO2) to be processed. Such a pre-conversion step allows for lower overall gas flows, smaller equipment and better overall heat recovery. As is common practice with modern acid plants excess heat is recovered as medium pressure steam which is used for power generation in a fully condensing turbine.

The conversion of sulphur dioxide to sulphur trioxide is catalyzed with a specialist vanadium pentoxide catalyst over four stages with interstage cooling. After the third stage the gas is passed through an intermediate adsorption tower for removal of most of the contained SO3. The final conversion stage ensures that overall conversion efficiency (SO2 to SO3) exceeds 99.9%. Gas exiting the last stage is passed through the final adsorption packed tower for removal of the remaining SO3.

Acid strength in the circulating acid system is maintained at 98.5% by addition of water and by bleeding acid at a rate equivalent to the production rate to the storage tanks.

Tail gas from the acid plant containing less than 400ppm SO2 is routed to the multipurpose desulphurization plant described in 1.3.10.5 above. After Desulphurization the tail gas contains less than 100 ppm SO2 well below generally accepted environmental limits

It appears that the acid plant does not have capacity to produce acid from sulphur dioxide arising from any Kamoa : Kakula ratio exceeding 70:30.

17.4.13 Slag Treatment plant

As noted in section 17.4.4 (slag cleaning furnace) above, SCF slag is delivered in pots to a slow cooling yard. The slow cooling process allows coalescence of remaining metallics and sulphides into larger sizes and for preferential crystallization of slag components.

Once the slag is solid it is tipped from the pots and then further cooled with water sprays. Cooled slag is broken up using a hydraulic hammer with broken slag being transferred to the slag treatment plant feed hopper.

A fairly standard comminution and flotation plant is used to recover copper metallics and sulphides from the slag. It consists of the following steps:

- Jaw crusher to reduce the slag to -150mm.
- Slag milling using a SAG / ball mill circuit with closed circuit cyclones on the ball mill. Grind size is 80% pass 45um.
- Rougher flotation in four 40m3 tank cells. Concentrate from the first cell reports directly to final concentrate.
- Scavenger flotation in four 40m3 tank cells.
- Rougher concentrate subject to single stage cleaning in two 16m3 tank cells to produce final concentrate.
- Scavenger concentrate subject to three stage cleaning in two banks of three 16m3 tank cells, concentrate reports to the single stage rougher concentrate cleaner.
- Overall recovery is 87% into a concentrate grading 22% copper. Discard slag contains less than 1.0% copper.
- Concentrate is thickened, filtered and transferred to concentrate storage
- Slag tailings are thickened and pumped to the Kakula final tailings tank for co-disposal to the tailings dam.

The slag recovery plant is designed for a throughput of 2080 tpd against a projected maximum slag rate of 2033 tpd. The design margin is only 2.5% and so there is little room for treating slag that may arise from Kamoa / kakula ratios of higher than 70:30.

17.4.14 Smelter utility requirements

The smelter requires the following utilities:

- Oxygen for the flash furnace.

- Nitrogen is required for blanketing duties mainly in the steam dryer and the coal pulverization and handling section.
- Demineralized water for the DBF waste heat boiler and the acid plant waste heat boilers.
- Process water for cooling duties and gas cleaning and slag retreatment.
- Electrical power consumption and production.

17.4.14.1 Oxygen plant

Oxygen consumption at maximum is about 35,000 NM3/hr (93% O2 basis).

A vacuum pressure swing adsorption (VPSA) plant is specified for the duty and has been designed to produce up to 38000 NM3/hr (93% O2 basis). A 10% design margin is thus in place.

The VPSA technology is 50% more power efficient than cryogenic technology given that there is no requirement for ultrapure oxygen. It is also simpler to operate and more flexible.

Early operation of the smelter suggests that the oxygen plant is not able to achieve design oxygen production rates and that there will be a need to install additional VPSA modules.

17.4.14.2 Nitrogen plant

Nitrogen consumption at maximum is as follows (99% N2 basis):

- Concentrate steam dryer 3000 Nm3/hr.
- Coal pulverization and transfer 2320 Nm3/hr.
- TOTAL 5320 Nm3/hr.

A pressure swing adsorption (PSA) plant is specified for the duty and has been designed to produce up to 11000 Nm3/hr (99% N2 basis). There is a healthy design margin of 50%.

17.4.14.3 Demineralized water

Total boiler water supply is 39.5 m3/hr. Most of the steam generated is condensed and recycled. It is supplied by two conservatively sized 30 m3/hr Demin water modules. To eliminate the use of reagents the plants consist of two stages of reverse osmosis followed by electrodeionization using ion exchange membranes.

17.4.14.4 Process water

A raw water treatment plant with a capacity of 800 m3/hr is provided to supply process water for a multitude of duties in the smelter. Raw water is supplied from underground dewatering or a wellfield. The water is treated using a combination of coagulation, settling and filtration. Treated water is stored first in a fire water pond and then a process water pond. Total pond capacity is 5800 m3. Process water pumps can deliver a maximum of 1300m3/hr to the smelter.

17.4.15 Power generation

High pressure steam from the DBF waste heat boiler is used in a high-pressure turbine to produce four MW of power. Low pressure steam from this turbine is combined with low pressure steam from the acid plant waste heat boilers. This is for two duties:

- To generate another four MW of power in a condensing turbine.

- For concentrate drying.

17.4.16 Smelter consumables and utilities

Table 17.5 below summarizes the important consumables and utilities. The list is not exhaustive as all the minor consumables are not included.

Table 17.5 Smelter consumables and utilities

Smelter Plant			**KAMOA COPPER SA**				
Consumables and Utilities			**KAMOA COPPER SMELTER PROJECT**				
				Date: March 2026			
Designation	**Unit**	**Total (/a)**			**Per ton copper anode**		
Concentrate mix		70% Kamoa	50% Kamoa	20% Kamoa			
Raw Material Input							
Copper Concentrate	t	1 203 506	1 111 613	998 053	2.41	2.22	2.00
Copper grade		42%	45%	50%			
Product Output							
Anode	t	500 000	500 000	500 000			
100% H_2SO_4	t	782 642	658 940	504 210	1.57	1.32	1.01
Slag tailings	t	742 264	657 829	549 975	1.48	1.32	1.10
Consumables							
Light Diesel Total	t	15 031	22 809	26 953	0.0301	0.0456	0.0539
Fine Coal (DBF)	t	0	0	0	0.0000	0.0000	0.0000
Fine coal Total	t	0	0	0	0.0000	0.0000	0.0000
Lime Total	t	57 636	67 996	75 396	0.115	0.136	0.151
Coke Total	t	23 315	18 373	11 879	0.0466	0.0367	0.0238
Limestone Total	t	49 129	48 764	48 363	0.098	0.098	0.097
Electric power							
TOTAL POWER	MWh	363 250	338 265	302 986	0.7265	0.6765	0.6060
Water							
Process water	10^3m^3	4425	4425	4425	0.0088	0.0088	0.0088
Demin water	10^3m^3	46	46	46	0.0001	0.0001	0.0001

18 Project infrastructure

18.1 Introduction

The Kamoa-Kakula Project requires additional mining and processing infrastructure for future operations to support the proposed Mineral Reserve plan. The additional infrastructure is to complement existing mining and mineral processing facilities which consist of the:

- Kakula Mine
- Kakula Concentrator
- Kakula Smelter
- Tailings Storage Facility No1
- Kansoko Mine
- Kamoa 1 Mine
- Kamoa Phase 3 Concentrator

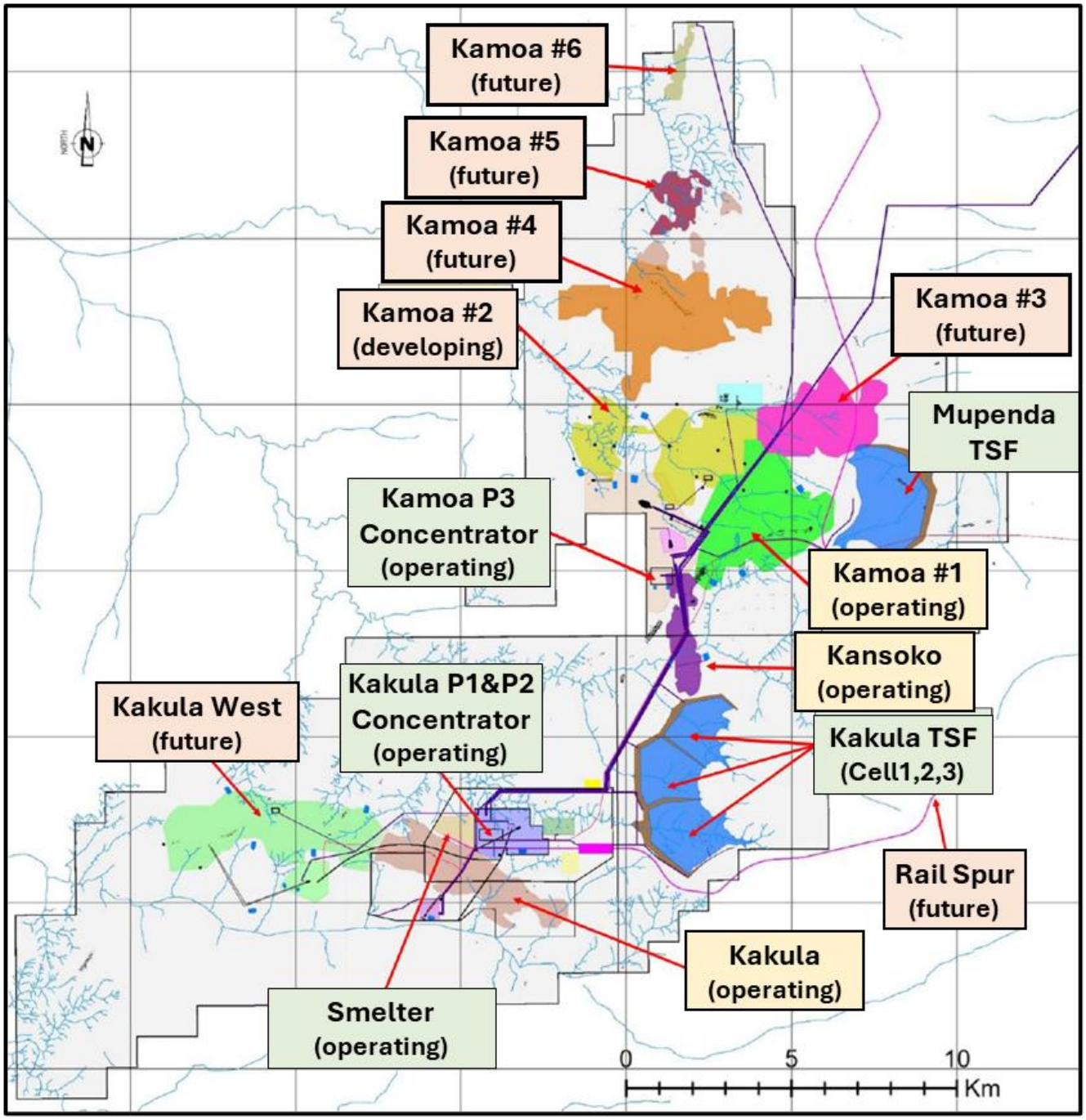
Included in the study is the infrastructure requirements for the existing mines, taking all the infrastructural development into account. The additional LOM mines included in the study are:

- Kakula West Mine
- Kansoko Sud Mine
- Kamoa 2 Mine
- Kamoa 3 Mine
- Kamoa 4 Mine
- Kamoa 5 Mine
- Kamoa 6 Mine

18.1.1 Site Plan and Layout – Overall

An overall blockplan showing the location of the mines, concentrators, smelter and tailings storage facilities is shown in Figure 18.1. The infrastructure required to support life-of-mine operations at each mine comprises of on-site and off-site facilities. On-site facilities consist of buildings, workshops and services infrastructure located at the mine's boxcut and will be established during mine access development. Off-site facilities are located remotely from the mine's boxcut area and are established over the life-of-mine across the orebody footprint. These facilities include ventilation infrastructure and auxiliary services associated with underground mining production.

Figure 18.1 Kamoa-Kakula Project Site Plan



Source: Ivanhoe 2025

18.2 Basis of Infrastructure Design, Selection and Sizing

The location, selection, preliminary design and sizing of various infrastructure facilities and equipment is based on the current design philosophy for the Kamo-Kakula Phase 3 execution project and inputs from Kamoa Copper SA project team and the following specialist consultants:

- AMC Consultants – Mine design and production scheduling.
- WSP – Underground water inflow modelling.

- Paterson & Cooke – Backfill plant design.
- Epoch Resources - Tailings storage facility design.
- Kamoa Power - Bulk electrical power supply.
- BBE – Underground mine ventilation and cooling design.

18.3 Kamoa-Kakula Surface Infrastructure Overview

The Kamoa-Kakula Surface infrastructure consist of all infrastructural requirements, to support the mining operations. The proximity of the proposed infrastructure and existing infrastructure was considered, to ensure that, where possible, existing services and infrastructure is utilized. This resulted in a reduction in the required infrastructure for the proposed mines. The same principle has been applied to the proposed mines to reduce the number of surface infrastructure complexes.

The infrastructure includes the roads, power supply and reticulation, water and sanitation, stormwater and dewatering control and management systems, acid water management systems (where applicable), ventilation and supporting infrastructure, buildings, backfilling and reticulation systems and tailings delivery lines and servitudes.

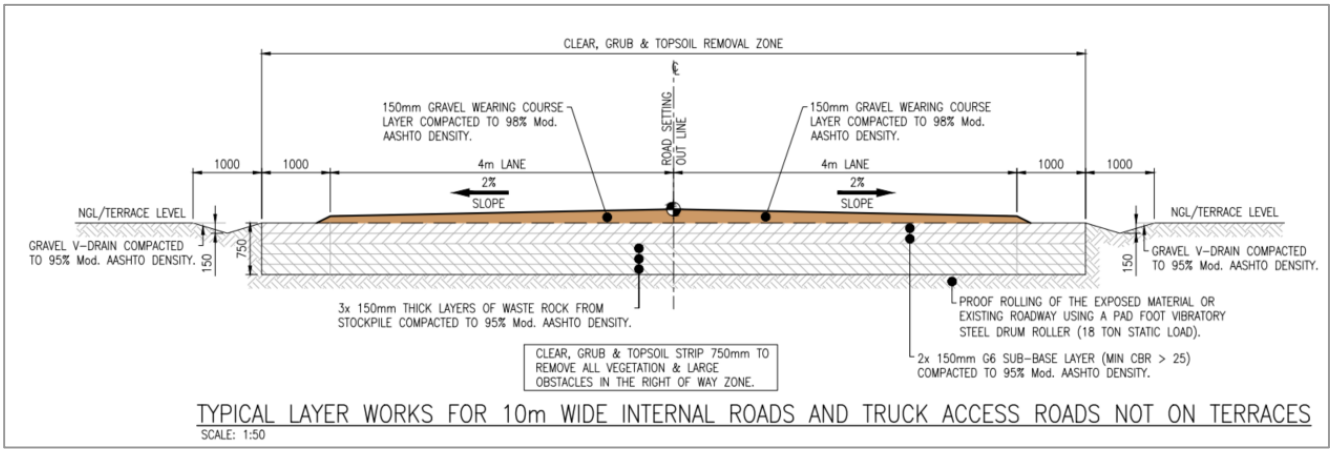
18.3.1 Roads

The roads can be separated into three categories namely main access, service and haul roads. The road designs are based on typical designs and as-built designs. No vertical and horizontal designs were completed during the PFS study.

18.3.1.1 Main Access Roads

The main access roads will connect the existing and the new mines. The road will be an extension and expansion of the existing roads network only. Refer to Figure 18.2, for the typical main access road section.

Figure 18.2 Main access road - typical section



Source: DRA 2026

The distance allowance for access roads associated with each mine is summarized in Table 18.1.

Table 18.1 Main Access Road – Distance allocation per mine

Item No	Description	Distance Allowance [km]	Comments
1	Kansoko Mine	N/A	Existing - No additional haul roads required
2	Kakula Mine	N/A	Existing - No additional haul roads required
3	Kamoa 1 Mine	N/A	Existing - No additional haul roads required
4	Kansoko Sud Mine	0.2	-
5	Kamoa 2 Mine	6.32	-
6	Kamoa 3 Mine	2.1	-
7	Kamoa 4 Mine	4.66	-
8	Kakula West Mine	4.67	-
9	Kamoa 5 Mine	0.25	-
10	Kamoa 6 Mine	0.2	-

Geometric Design

The Main Access Road is designed as a two-lane, single carriageway gravel road. The total width of the road is 10 m. Each lane is 5 m wide including the road shoulder. The vertical and horizontal alignment was not done for this level of study.

Pavement Design

The main access road will consist of a gravel access road that is designed in accordance with TRH 4. Refer to Table 18.2 for the design of the pavement layers.

The typical pavement design life is 15 years.

Table 18.2 Main Access Road – Typical Layer works

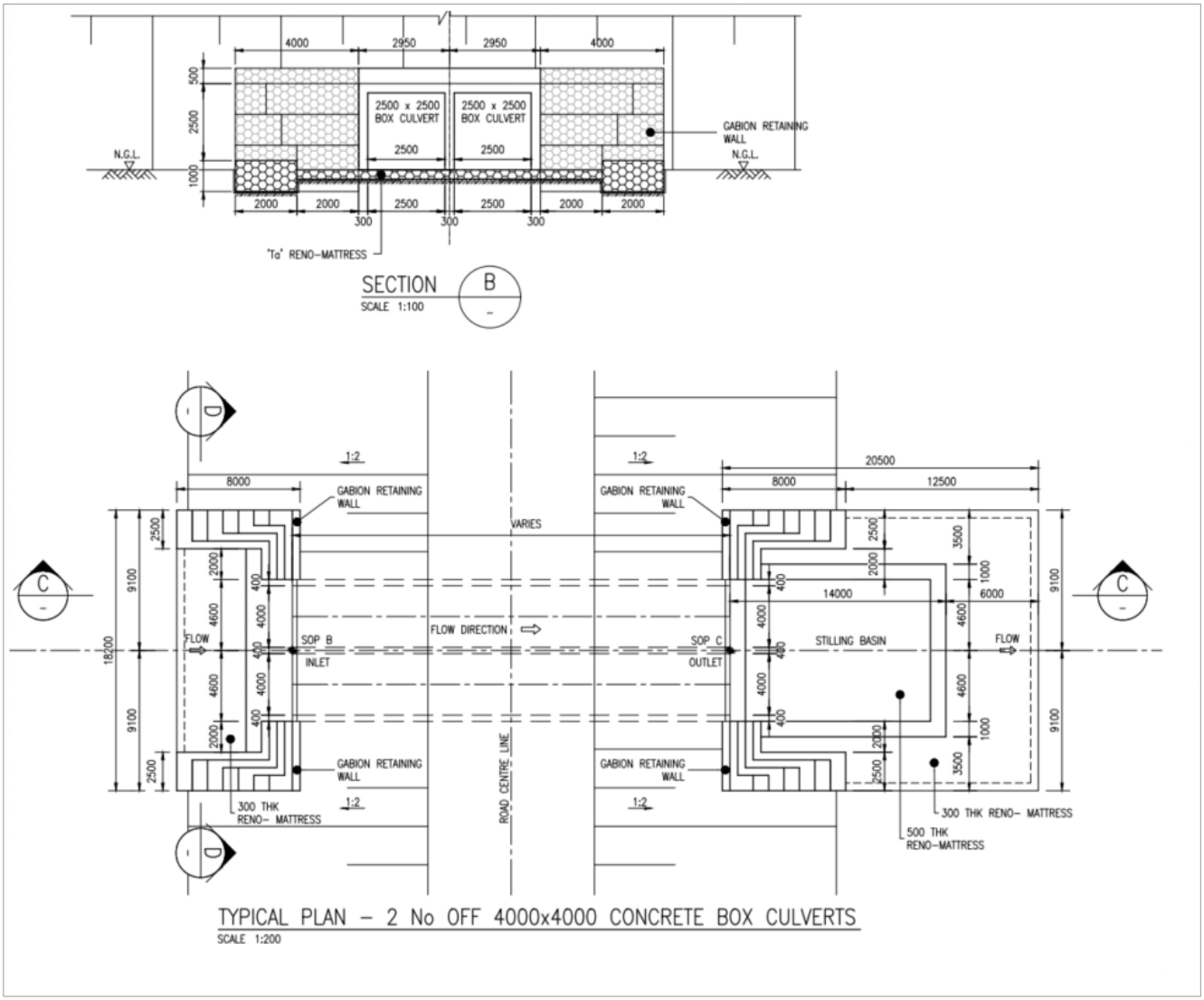
Layer No.	Description	TRH 14 Material Type	Compaction
1	150 mm thick gravel wearing course	G5	98% MOD. AASHTO
2	300 mm thick sub-base (upper and lower selected layer)	G6	95% MOD. AASHTO
3	450 mm thick waste rock fill (ground replacement layer))	N/A	95% MOD. AASHTO

Road Stormwater Management and Drainage

Stormwater management for the main access road is limited to river or stream crossings. All stormwater crossings are designed for a 1:10-year storm event. Figure 18.3 and Figure 18.4 shows a typical culvert section and outlet structure.

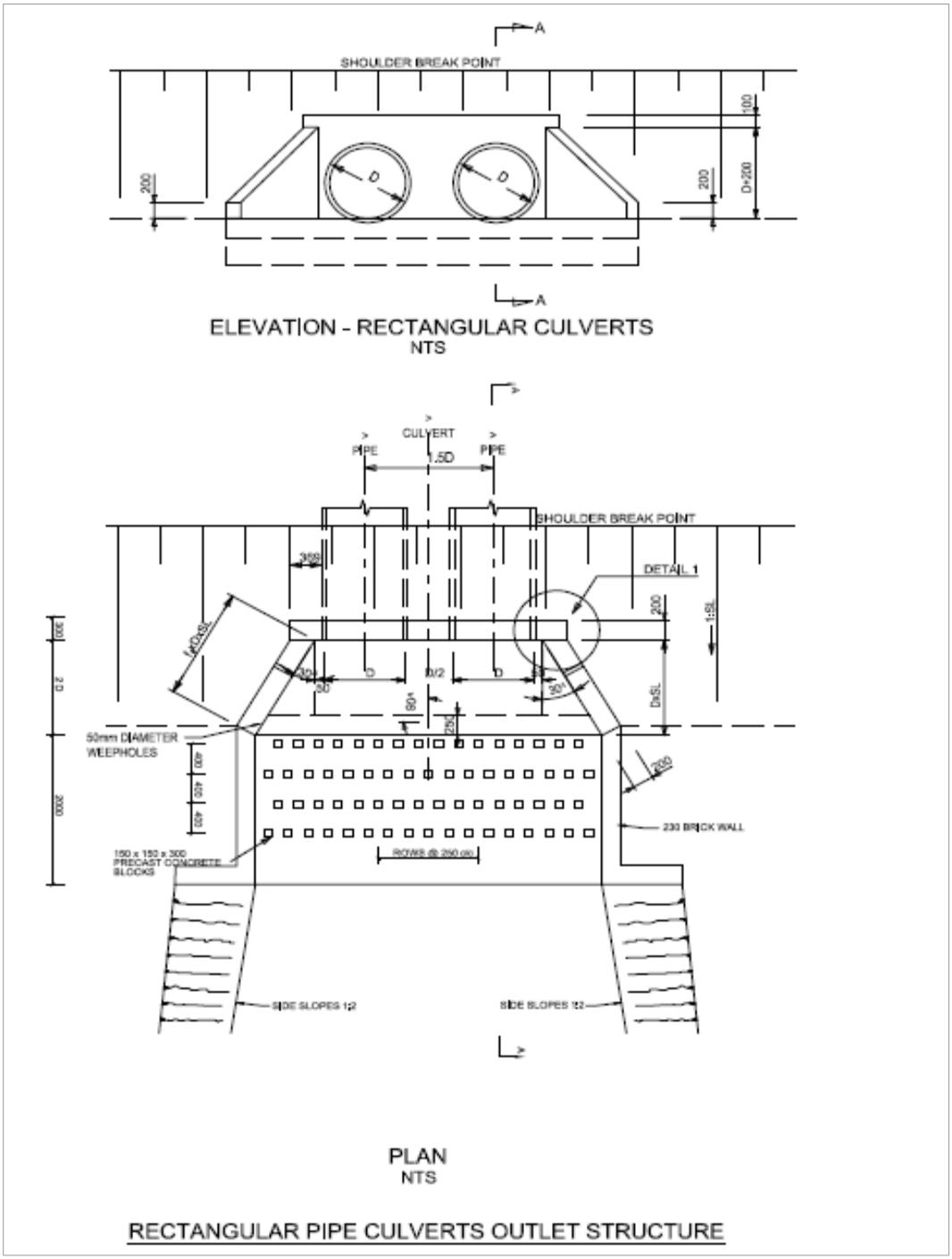
Based on the existing road construction, no allowance was made for any side or toe drains.

Figure 18.3 Main Access Road - Typical Stormwater Culvert



Source: DRA 2026

Figure 18.4 Typical Culvert Details



Source: DRA 2026

It should be noted that no pre-cast culverts are available and needs to be casted in situ. Pipe culverts would be constructed with HDPE pipes.

Safety Measures

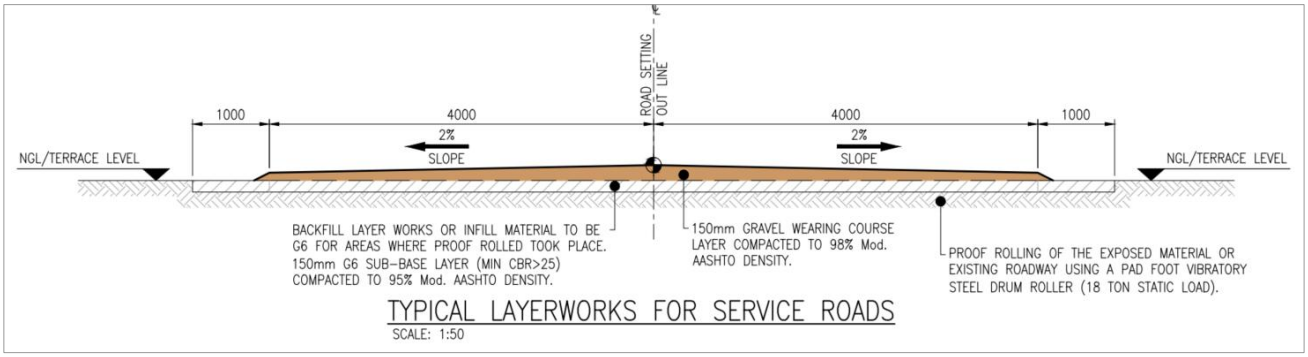
No allowance has been made for safety berms and/or guardrails because the road is close to ground level. The necessary dust suppression or mitigation measures are expected to be in place as part of maintenance and on-going operations.

18.3.1.2 Service access roads

The service roads consist of minimal earthworks and serve as construction and maintenance access to the different “remote” infrastructure. These roads are linked with the existing and proposed haul and main access roads.

The typical design and allowance for service roads per mine is shown in Figure 18.5 and tabled for the various mines in Table 18.3.

Figure 18.5 Service roads - typical section



Source: DRA 2026

Table 18.3 Service access road – distance allocation per mine

Item No	Description	Distance Allowance (km)	Comments
1	Kansoko Mine	0.45	Existing – Minor additional roads required
2	Kakula Mine	N/A	Existing - No additional roads required
3	Kamoa 1 Mine	N/A	Existing - No additional roads required
4	Kansoko Sud Mine	5.59	-
5	Kamoa 2 Mine	15.72	-
6	Kamoa 3 Mine	21.05	-
7	Kamoa 4 Mine	22.84	-
8	Kakula West Mine	44.69	-
9	Kamoa 5 Mine	16.76	-
10	Kamoa 6 Mine	7.06	-

Geometric design

The service road is designed as a single-lane gravel road. The total width of the road is 5 m wide including the road shoulder.

Pavement design

The service road will consist of a gravel access road that is designed in accordance with TRH 4. Refer to Table 18.4 for the design of the pavement layers.

Table 18.4 Service road – typical layer works

Layer No.	Description	TRH 14 Material Type	Compaction
1	150 mm thick gravel wearing course	G5	98% MOD. AASHTO
2	150 mm thick sub-base (upper and lower selected layer)	G6	95% MOD. AASHTO

Road stormwater management

No allowance for stormwater was allowed for. Maintenance will be ongoing as and when required. Major and minor river crossings will be part of the main access and haul road designs.

Safety measures

No safety berms and safety measures were allowed for, as these roads are not public roads and are mainly access for mine-related inspections and maintenance.

18.3.1.3 Haul roads

Haul roads are defined as roads that are specifically provided for the use of off-highway trucks and mining machinery (vehicles that are not allowed for use on public roads) and are typically used in mining activities. However, the hauling fleet for the project will consist of road legal tipper trucks and the haul road will be a dedicated access road, for the hauling of materials.

The distances for the haul roads are divided between the various mines as tabled in Table 18.5.

Table 18.5 Haul road – distance allocation per mine

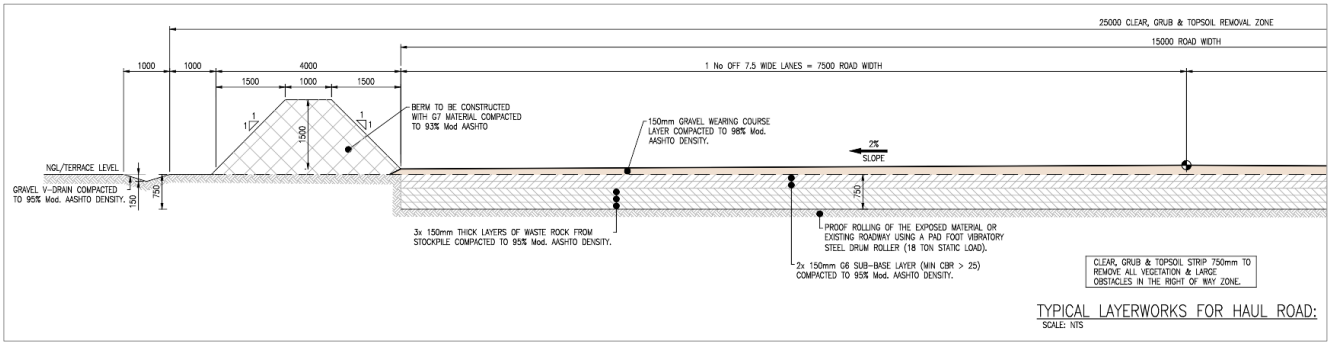
Item No	Description	Distance Allowance	Comments
1	Kansoko Mine	N/A	Existing - No additional haul roads required
2	Kakula Mine	N/A	Existing - No additional haul roads required
3	Kamoa 1 Mine	N/A	Existing - No additional haul roads required
4	Kansoko Sud Mine	1.39	-
5	Kamoa 2 Mine	20.01	Includes the haul road connection between Kamoa mine and the Kakula concentrator
6	Kamoa 3 Mine	7.35	-
7	Kamoa 4 Mine	13.25	-
8	Kakula West Mine	10.84	-
9	Kamoa 5 Mine	3.56	-
10	Kamoa 6 Mine	7.78	-

Geometric Design

The haul road is designed as a four-lane, dual direction, single carriageway with a gravel capping. The total width of the road is 15 m excluding berms and batters. A maximum design speed limit of 60 km/h was used as the design criterion, with gradients not exceeding 10%. Geometric standards (both horizontal and

vertical) are designed to accommodate road tipper trucks or similar haul trucks. Refer to Figure 18.6 for the typical haul road section.

Figure 18.6 Typical Haul Road Section



Source: DRA 2026

Pavement Design

The camp access road will consist of a gravel access road that is designed in accordance with TRH 4. Refer to Table for the design of the pavement layers.

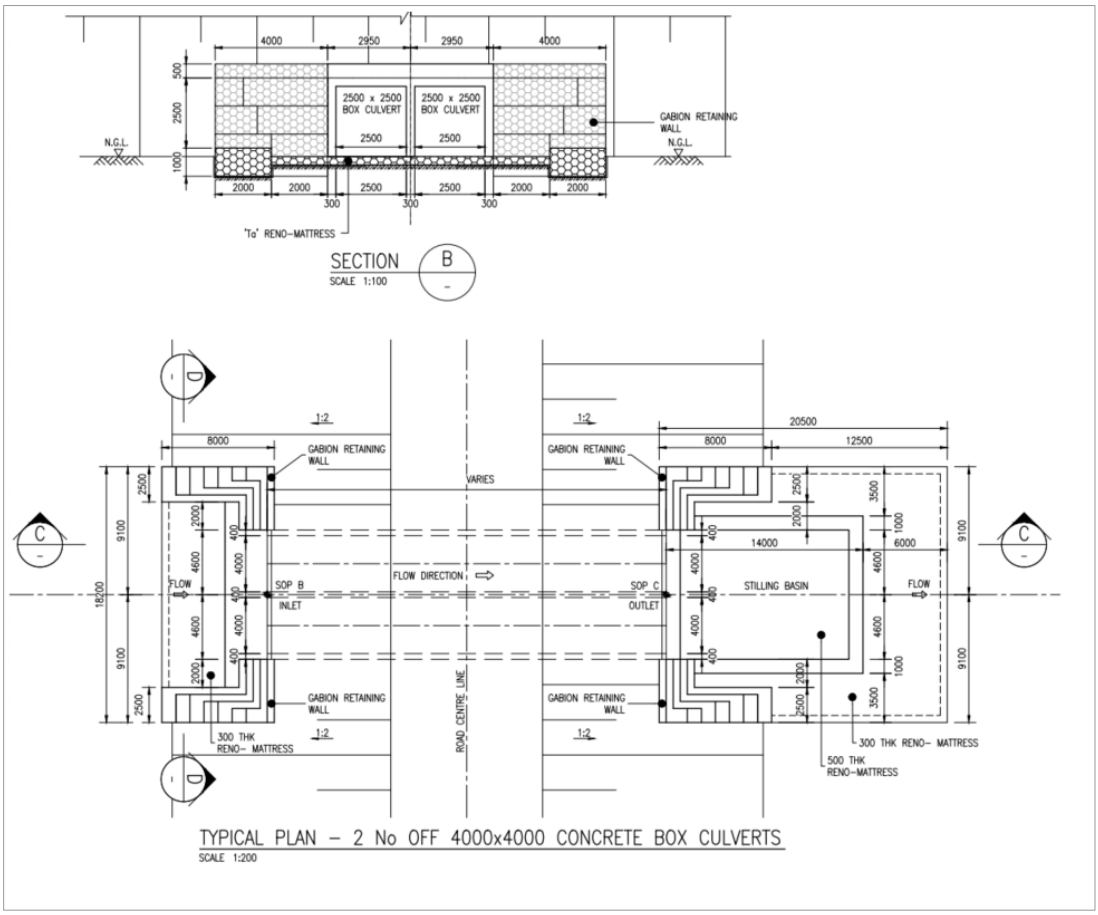
Table 18.6 Haul Road – Pavement Design

Layer No.	Description	TRH 14 Material Type	Compaction
1	150 mm thick gravel wearing course	G5	98% MOD. AASHTO
2	300 mm thick sub-base (upper and lower selected layer)	G6	95% MOD. AASHTO
3	450 mm thick waste rock fill (ground replacement layer))	N/A	95% MOD. AASHTO

Road Stormwater Management

Stormwater management for the haul road consist of river or stream crossings and an earth V-drain on the toe of the safety berm. All stormwater crossings are designed for a 1:10-year storm event. Figure 18.7, shows a typical culvert section and outlet structure and Figure 18.8 shows the toe-drain detail next to the safety berm.

Figure 18.7 Haul Road Culvert – Typical Detail

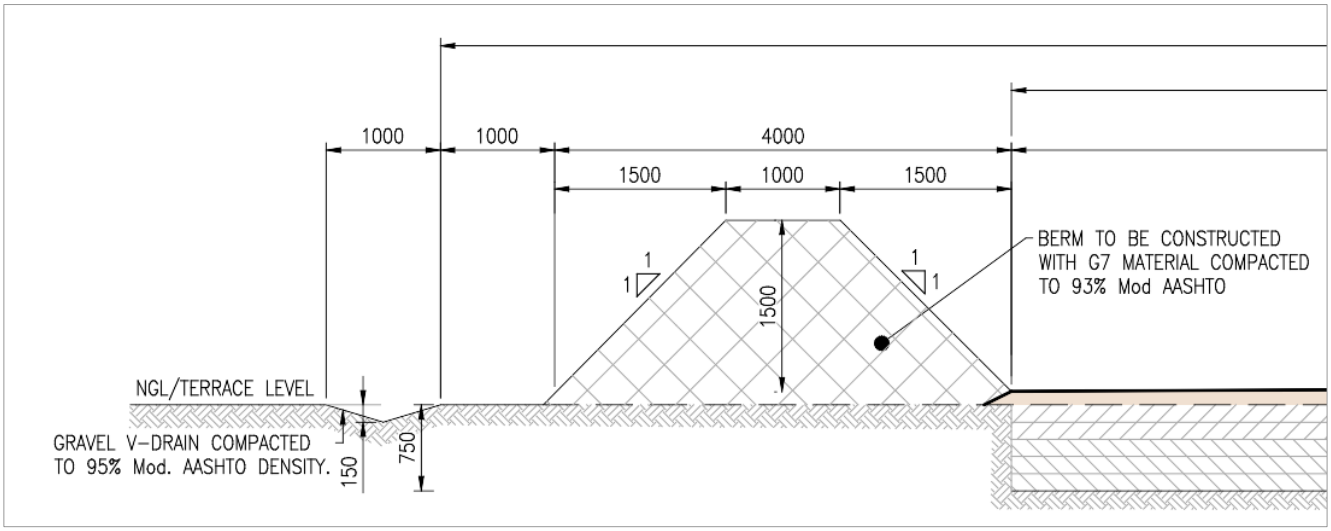


Source: DRA 2026

Safety Measures

Safety and precautionary measures have been included in the road design. These measures include safety berms that are at least half the height of the largest wheel travelling on the haul road. Figure 18.8 shows the minimum safety berm requirements.

Figure 18.8 Safety Berm and Toe-drain detail



Source: DRA 2026

18.3.2 Bulk Earthworks

18.3.2.1 Geotechnical Investigation

No geotechnical investigations were undertaken as part of the current MRMR Upgrade study. Based on previous completed geotechnical investigations for the existing mines, assumed geotechnical conditions were developed for the following structures:

- Roads
- Terraces and platforms
- Box cuts (Refer to Section
- Dewatering and services columns
- Buildings and Structures
- Ventilation shafts and collars

No geotechnical information is available for the new mines and services.

18.3.2.2 Typical Foundation

Due to the limited availability of site-specific geotechnical information, the earthworks designs were developed based on existing mine geotechnical data. Consequently, no detailed or structure-specific designs or modelling were undertaken at this stage.

18.3.2.2.1. Lightly Loaded Structures (< 150 kPa)

Most of the lighter structures are anticipated to be supported on shallow foundations, including spread footings and strip footings, and/or raft foundations founded on competent native soils and engineered crushed rock fill.

Based on the available geotechnical design, it was assumed that the first 0.5 m of the in-situ material would be unsuitable and need to be replaced with an engineered fill.

18.3.2.2.2. Medium to Heavily Loaded Structures (> 150 kPa)

For the medium to heavily loaded structures, the in-situ ground conditions are deemed unsuitable, and localized ground improvement, consisting of engineered fill, will be required. The excavation depths and engineered fill thicknesses will be determined from the loading, founding depth, and allowable settlement parameters.

Based on the available geotechnical information, it was assumed that the first 3 m of the in-situ material would be removed and replaced with engineered fill.

18.3.3 Terraces, Platforms and Stockpiles

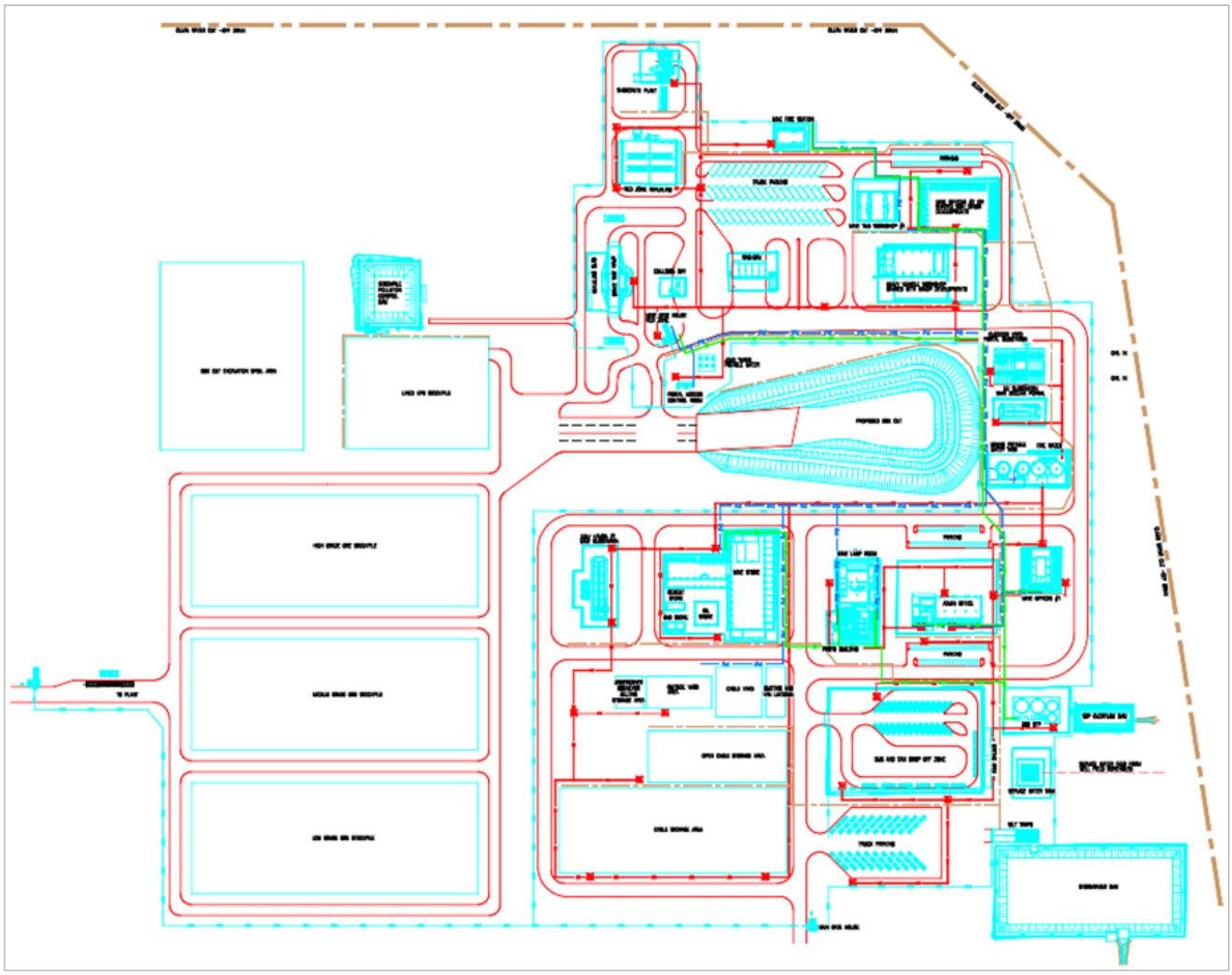
18.3.3.1 Terraces

The terraces and the required surface infrastructure was categorized as either “Major” or “Minor” surface infrastructure complexes as related to the required box cuts. The selection was based on the nature and extent of existing or proposed surface infrastructure and their relative proximity.

Major Infrastructure Complex

The major infrastructure complexes will house all major infrastructure, offices, workshops, stores etc. located at the different mining box cuts. The major infrastructure complexes will be strategically placed to service the minor infrastructure complexes and support surface infrastructure. For the major box cut infrastructure layout, refer to Figure 18.9.

Figure 18.9 Illustrative - Typical Major Box Cut Surface Infrastructure Area – Plan View

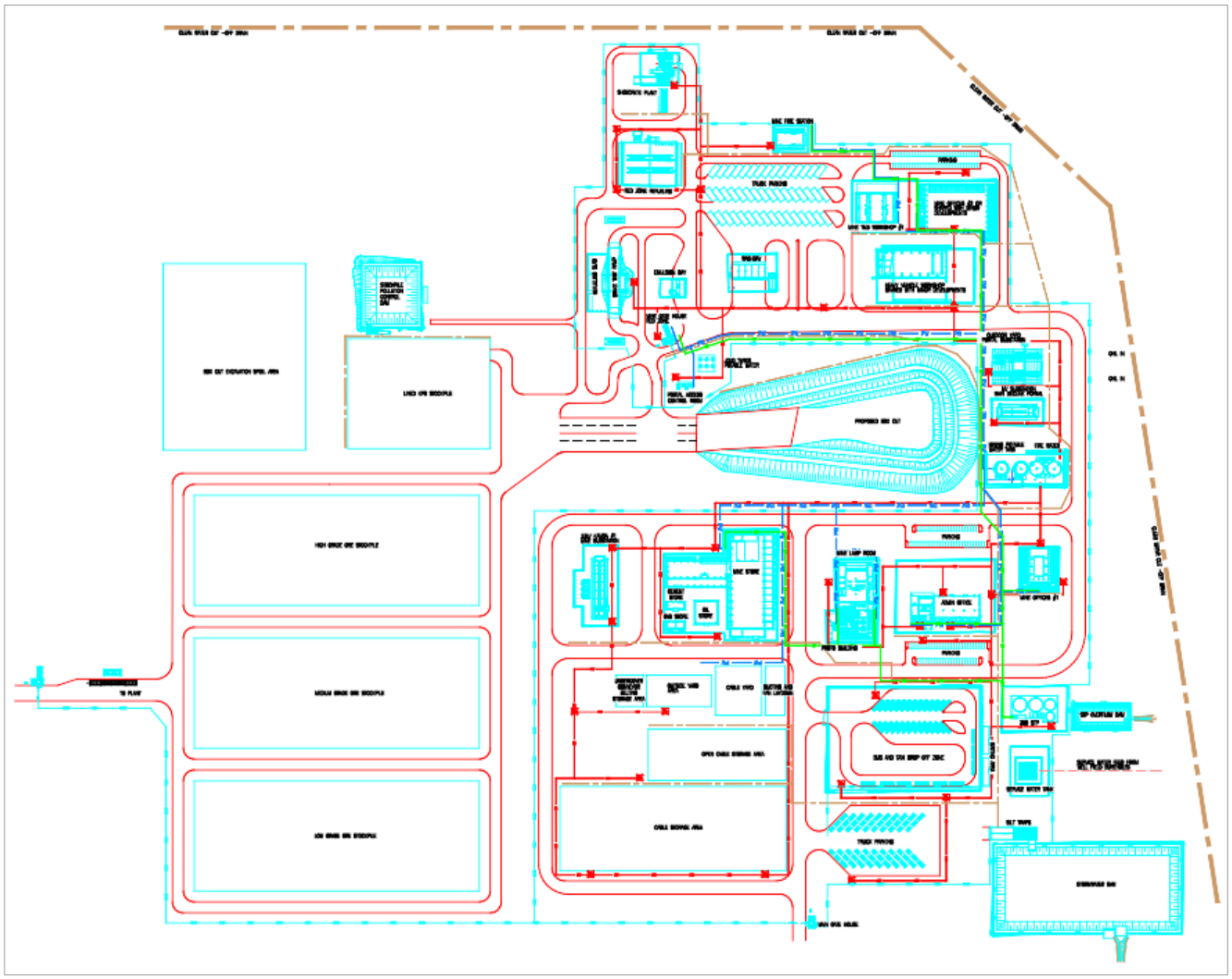


Source: DRA 2026

Minor Infrastructure Complex

The minor infrastructure complexes are designed to accommodate all supporting facilities, including offices, workshops, and storage areas, associated with the various mining box cuts, as outlined in Table 18.7. For the minor box cut infrastructure layout, refer to Figure 18.10

Figure 18.10 Illustrative - Typical Minor Box Cut Surface Infrastructure Area – Plan View



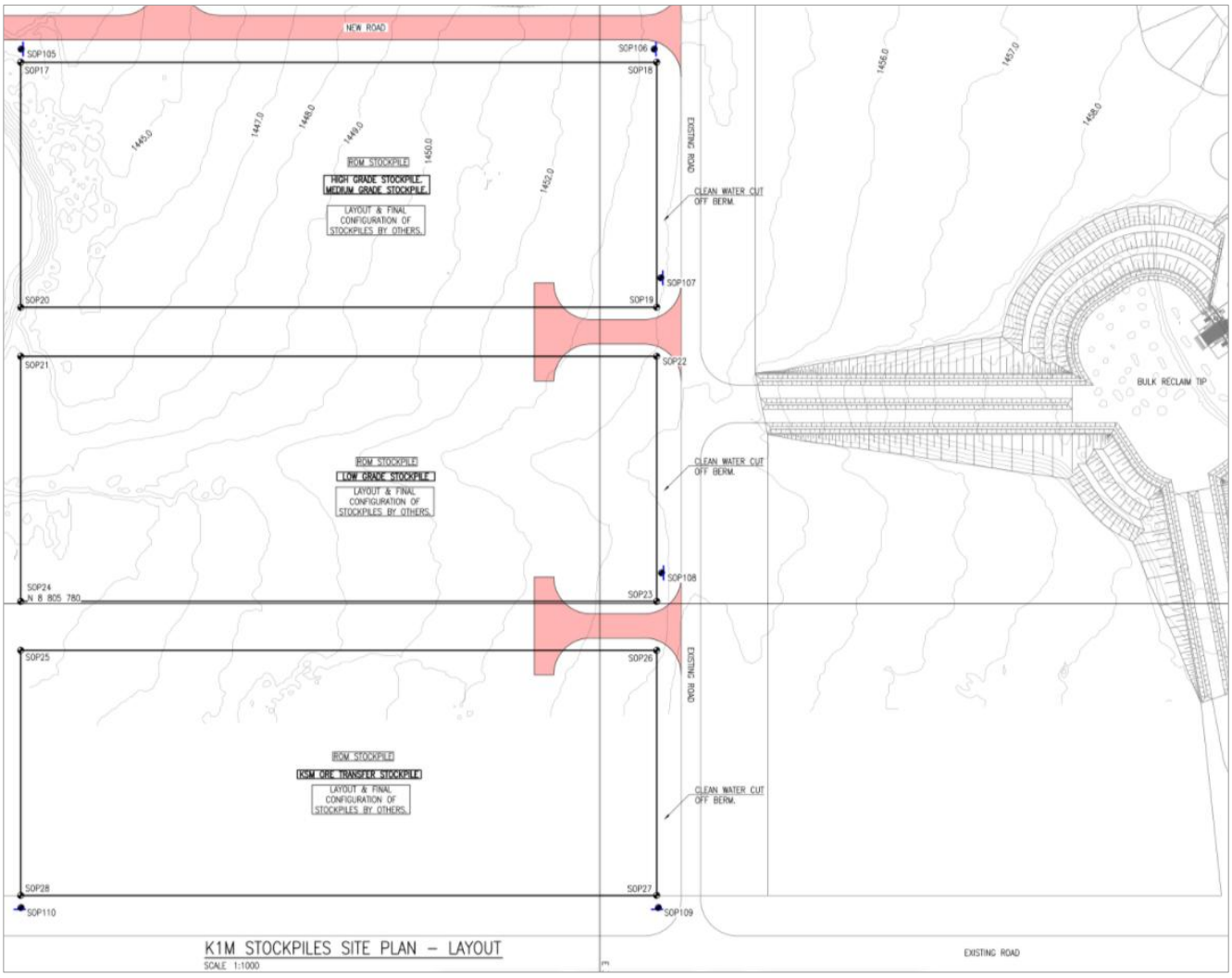
Source: DRA 2026

ROM Stockpile Area

Some of the proposed mining areas will not require the development of dedicated major or minor infrastructure terraces. This is primarily due to their close proximity to existing or planned box cuts that are already equipped with the necessary supporting infrastructure, including processing, handling, and service facilities. By leveraging these nearby established areas, duplication of infrastructure can be avoided, resulting in both capital cost savings and a reduced construction footprint.

In such cases, the affected box cuts will be limited to the provision of Run of Mine (ROM) stockpile areas only. These will typically consist of separate High, Medium, and Low-grade stockpiles, designed to allow for effective material segregation and grade control prior to processing. The sizing and layout of these stockpiles will be determined based on the mine's production schedule, haulage logistics, and blending requirements, ensuring operational efficiency while maintaining flexibility in ore handling. Refer to Figure 18.11 for a plan view of the typical stockpile terrace layout.

Figure 18.11 Typical Box cut ROM Stockpile Layout - Plan View



Source: DRA 2026

Infrastructure allocations

The allocation of mining complexes into Major and Minor categories is presented in Table 18.7. This classification was determined in consultation with the Kamoa Copper SA project team, considering the functional requirements of each box cut and its proximity to other supporting infrastructure. As a result, the provision of major infrastructure at every box cut was not deemed necessary. The allocation is not applicable for existing mines.

Table 18.7 Infrastructure Allocation per Mine

Item No	Description	Major Mining Infrastructure	Minor Mining Infrastructure
1	Kansoko Mine	Not Applicable	Not Applicable
2	Kakula Mine	Not Applicable	Not Applicable
3	Kamoa 1 Mine	Not Applicable	Not Applicable
4	Kansoko Sud Mine	-	Minor
5	Kamoa 2 Mine:		
5.1	Kamoa 2 East Mine	Not Applicable	Not Applicable
5.2	Kamoa 2 West Mine	-	Minor
6	Kamoa 3 Mine	-	Minor
7	Kamoa 4 Mine		
7.1	Kamoa 4 North Mine	-	Minor
7.2	Kamoa 4 South Mine	-	Minor
7.3	Kamoa 4 West Mine	Major	-
8	Kakula West Mine	Major	-
9	Kamoa 5 Mine	-	Minor
10	Kamoa 6 Mine	-	Minor

18.3.4 Stormwater Management

Typical stormwater management measures were determined for the different mining infrastructure complexes. This includes the stormwater management in and around each box cut. In principle the design separates contact (dirty) and non-contact (clean) water and will be applicable across all the mines.

The design is based on the following criterion:

- A 1:100-year flood line is applied, and all structures on the mine will be located outside of, or protected against, this flood level.
- A 1:50-year storm event was used to calculate the stormwater runoff and peak flows, to size the required stormwater 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 in water storage dams. Freeboard can be defined as the vertical distance between the full supply level (spillway crest level), and the lowest point on the dam wall crest.

Environmental protection and the reduction of potential surface water contamination included the following 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 required.

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 mobilized 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.
- Regular desilting of stormwater ponds will be required to retain storage capacity.
- Regular clearance of vegetation in stormwater channels will be required.
- Channels will require scour protection (heavy-duty geosynthetic filter and riprap or gabion mattresses) and regular maintenance where high flow velocity is expected.
- 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 settled 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 stormwater model, other than riprap scour protection.

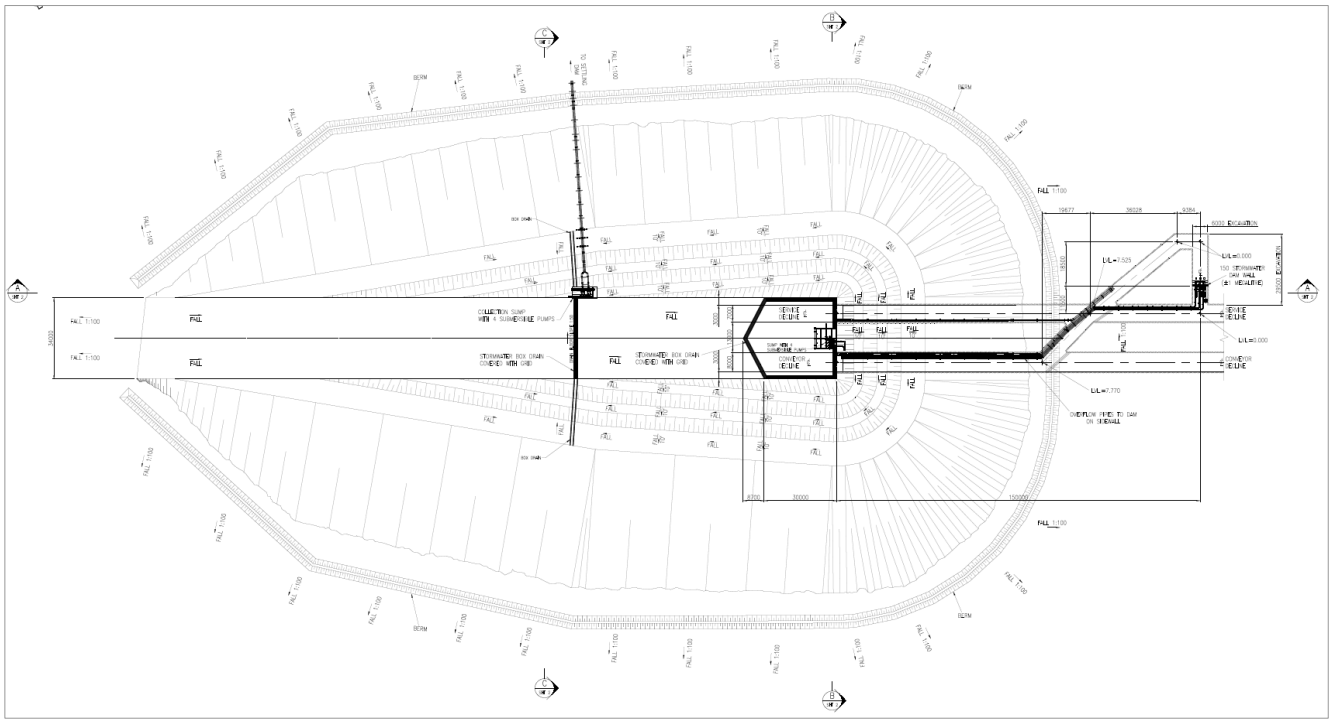
18.3.4.1 Box Cut Stormwater Management

Infrastructure for the management of surface runoff in the box cut, access portals, decline shafts and mining return water (from mining operations and ground water inflows) comprise sumps and transfer dams equipped with pumps, for the catchment and pumping of water to surface settling dams. The dirty water is treated by settling and removing sediment, adding lime for pH correction and in some instances, treated in a water treatment plant for re-use or disposal of excess water to the environment. The following facilities will be constructed for each mine:

- Two box cut sump pumpstations, each equipped with four (4) nr. 90 kW submersible pumps. The two sump pumpstations have a peak pumping capacity of 960 l/s for handling runoff in the box cut area.
- A stormwater dam and pumpstation, located in the main access decline, approximately 150 m from the highwall portal for capturing runoff into the decline shafts (“150 Stormwater Dam”). The associated pumpstation will be equipped with three (3) pump trains with a peak pumping capacity of 600 l/s.
- A decline bottom dam and pumpstation with similar pumping arrangement and pumping capacity to the 150 Stormwater Dam Pumpstation.

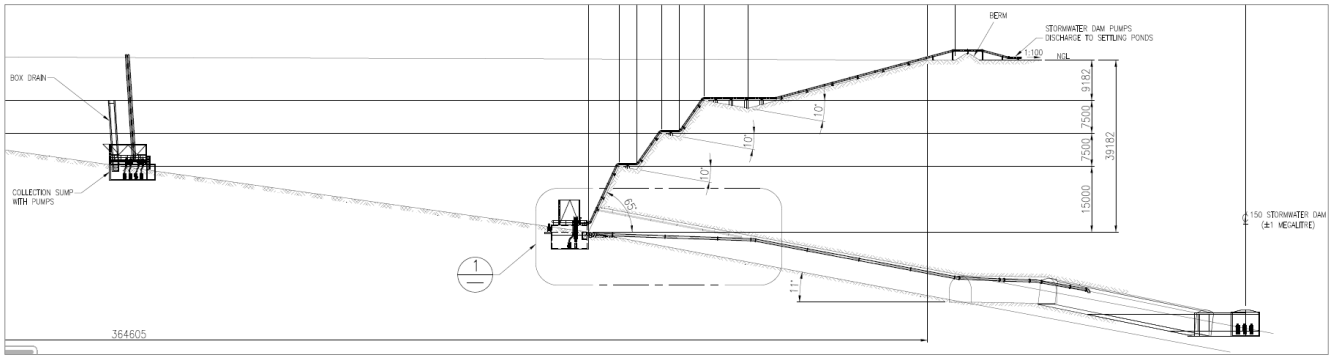
The three pumpstations discharge sediment laden water into a settling dam prior to the settled water discharging to the environment. Figure 18.12and Figure 18.13 illustrate the two sump pumpstations in the box cut and the 150 Stormwater dam and pumpstation located adjacent to the service declines.

Figure 18.12 Typical Box Cut Stormwater Management System - Plan View



Source: DRA 2026

Figure 18.13 Typical Box Cut Stormwater Management System - Section View



Source: DRA 2026

18.3.4.2 Contact and Non-contact Stormwater Management Systems

The stormwater management system is to ensure that non-contact water remain uncontaminated and that the contact water is captured in the pollution control facilities.

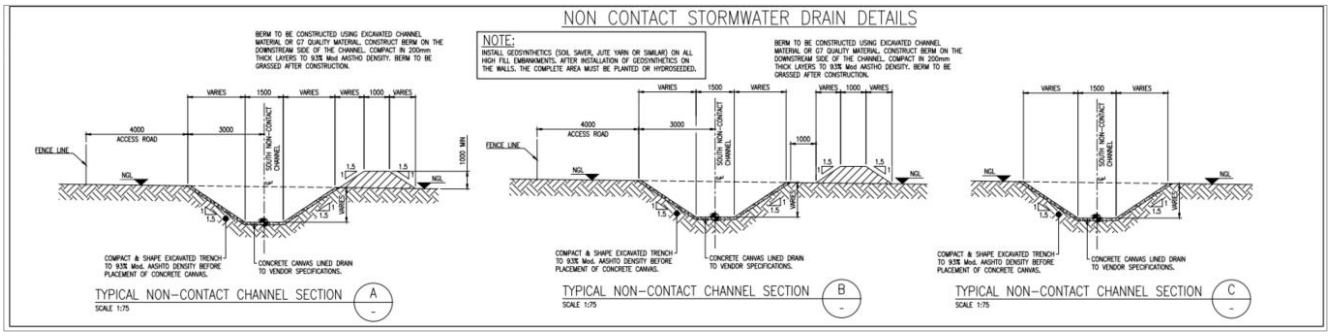
Non-contact Water

Cut-off Drains

A clean water cut-off drain is included, upstream of the proposed infrastructure. This is to ensure that during a storm event, non-contact runoff water will be collected and directed past the terrace and will remain uncontaminated.

Refer to Figure 18.14 for the typical non-contact stormwater drain details.

Figure 18.14 Typical Non-Contact Drains



Source: DRA 2026

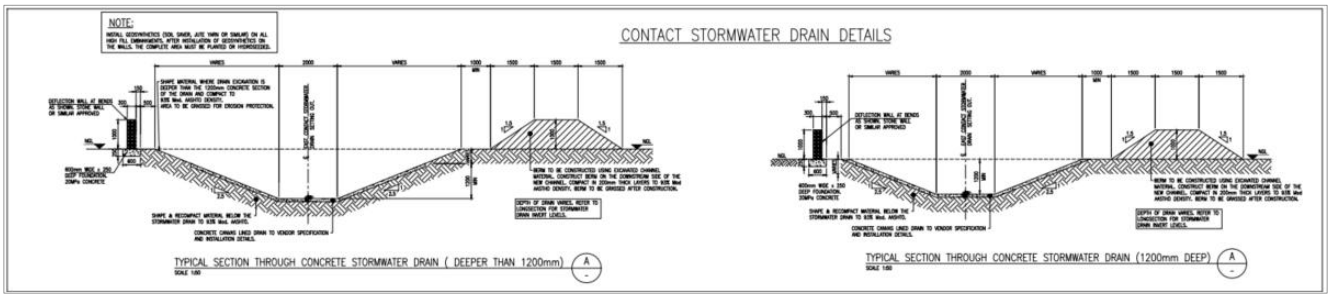
Contact Water

Cut-off Drains

All runoff water from the terrace will be considered dirty, or contact water, and will be collected and transferred to the pollution control dam (PCD) via concrete lined drains from where it will be re-used for mining and processing water demand.

Refer to Figure 18.15 for the typical contact stormwater drain detail.

Figure 18.15 Typical Contact Drain Detail

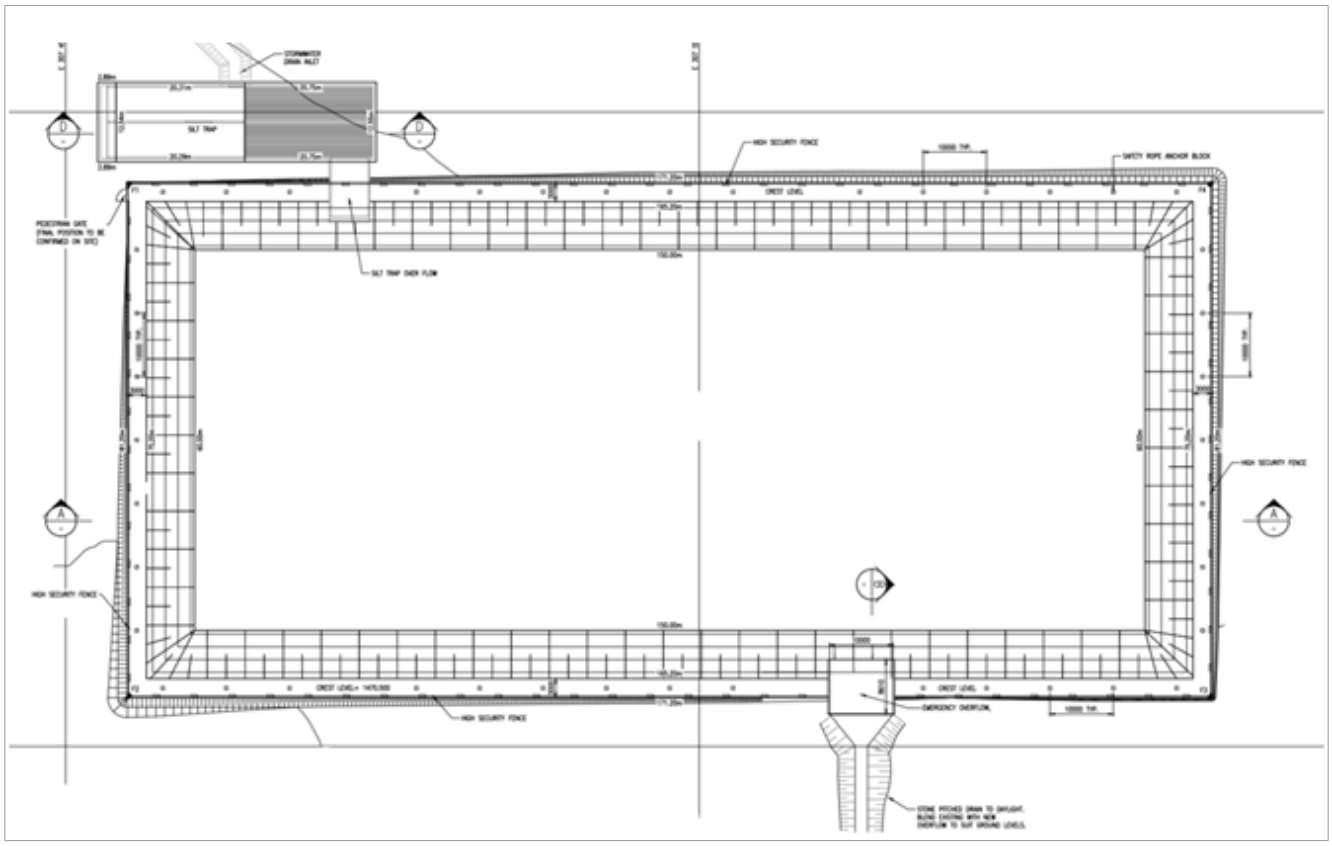


Source: DRA 2026

Pollution Control Dam (PCD)

The pollution control dams consist of HDPE lined earth dams, with required overflows and safety features as well as upstream sediment control facilities. All the dirty (contact) runoff water from the terraces and stockpiles will report to the pollution control facilities. Refer to Figure 18.16 for a typical PCD layout with silt trap and overflow arrangements.

Figure 18.16 Typical Pollution Control Dam

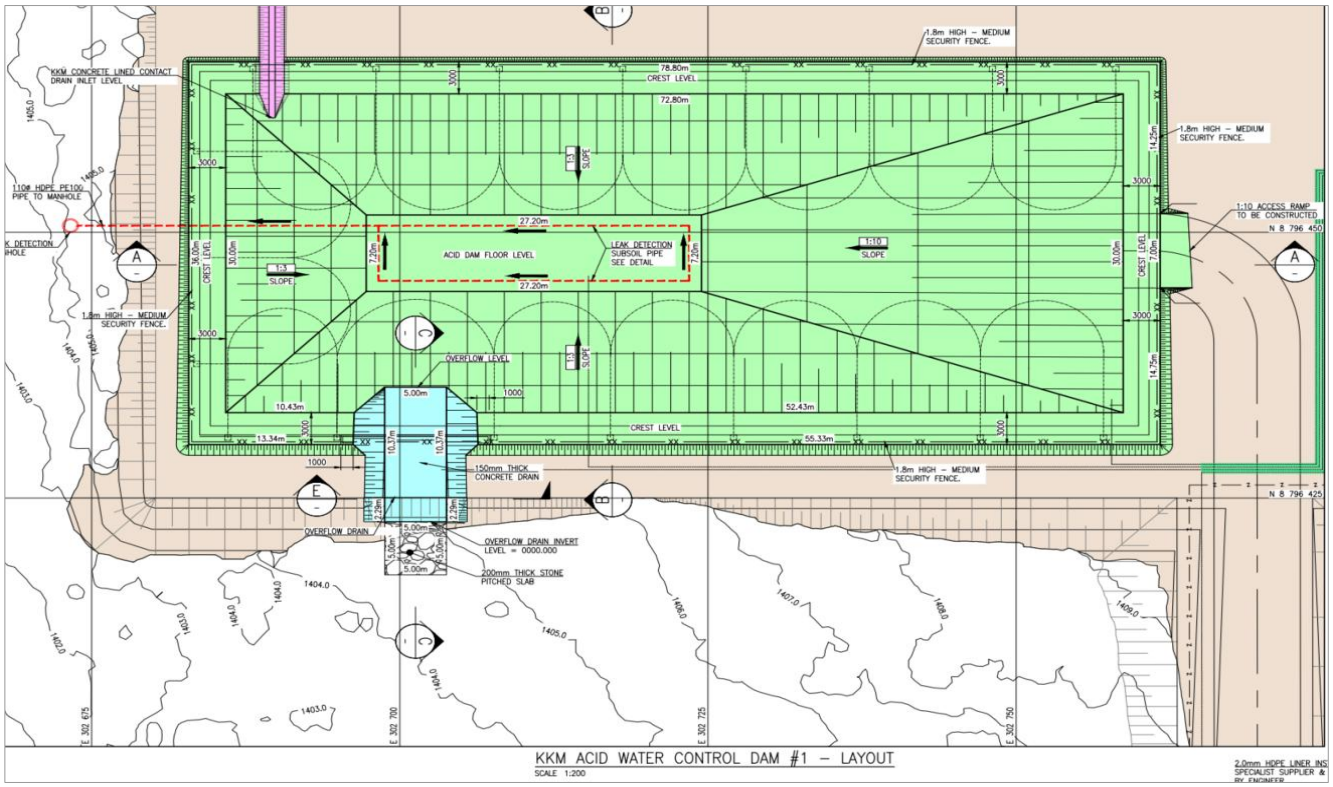


Source: DRA 2026

Acid Water Handling

It is anticipated that there will be a requirement to stockpile acid generating Kamoa Pyritic Siltstone (KPS) Waste material, during mine development. Temporary stockpile and pollution control system have been considered with a permanent acid control waste stockpile to be located at the existing Kamoa 1 Mine area. Refer to Section 18.9.3.1 for the centralized acid mine waste stockpile and runoff water treatment facilities. Refer to Figure 18.17 for typical acid water control dam whilst Figure 18.18 and Figure 18.19 address the proposed KPS Waste Stockpiles.

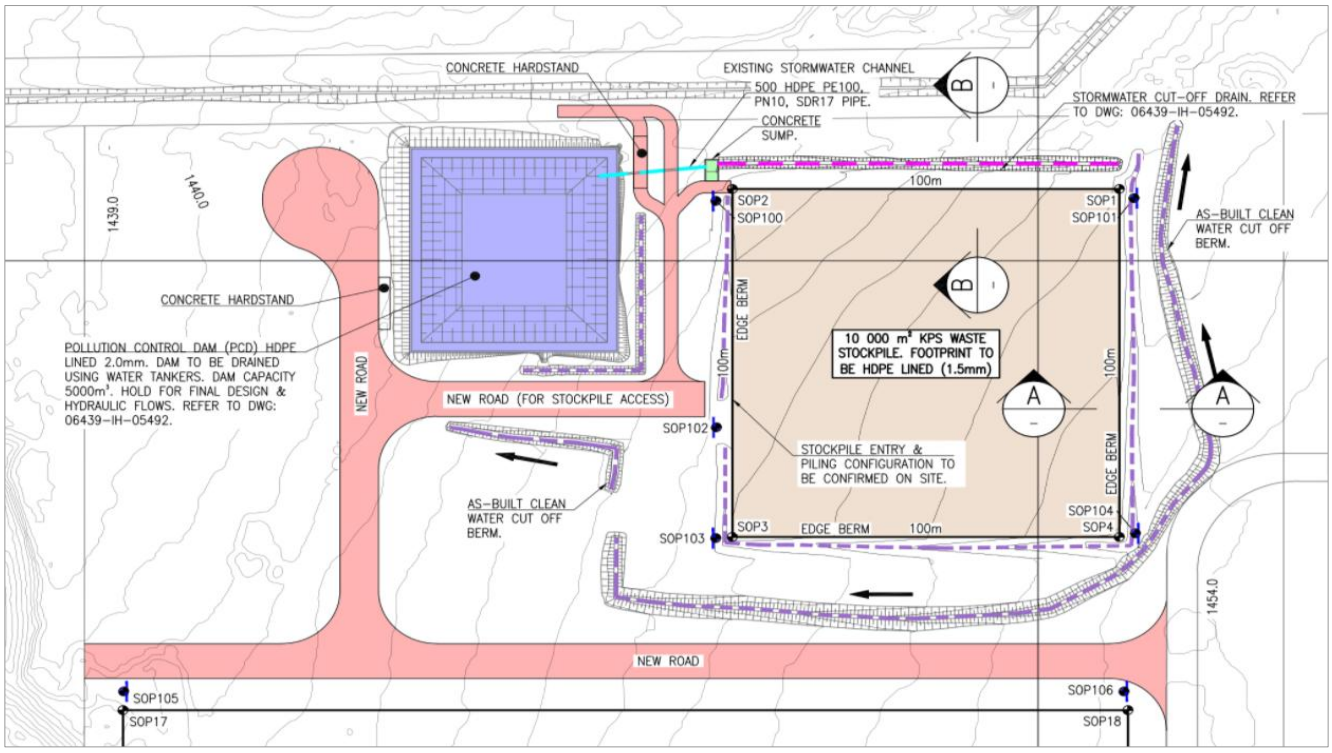
Figure 18.17 Typical Acid Water Control Dam - Plan View



Source: DRA 2026

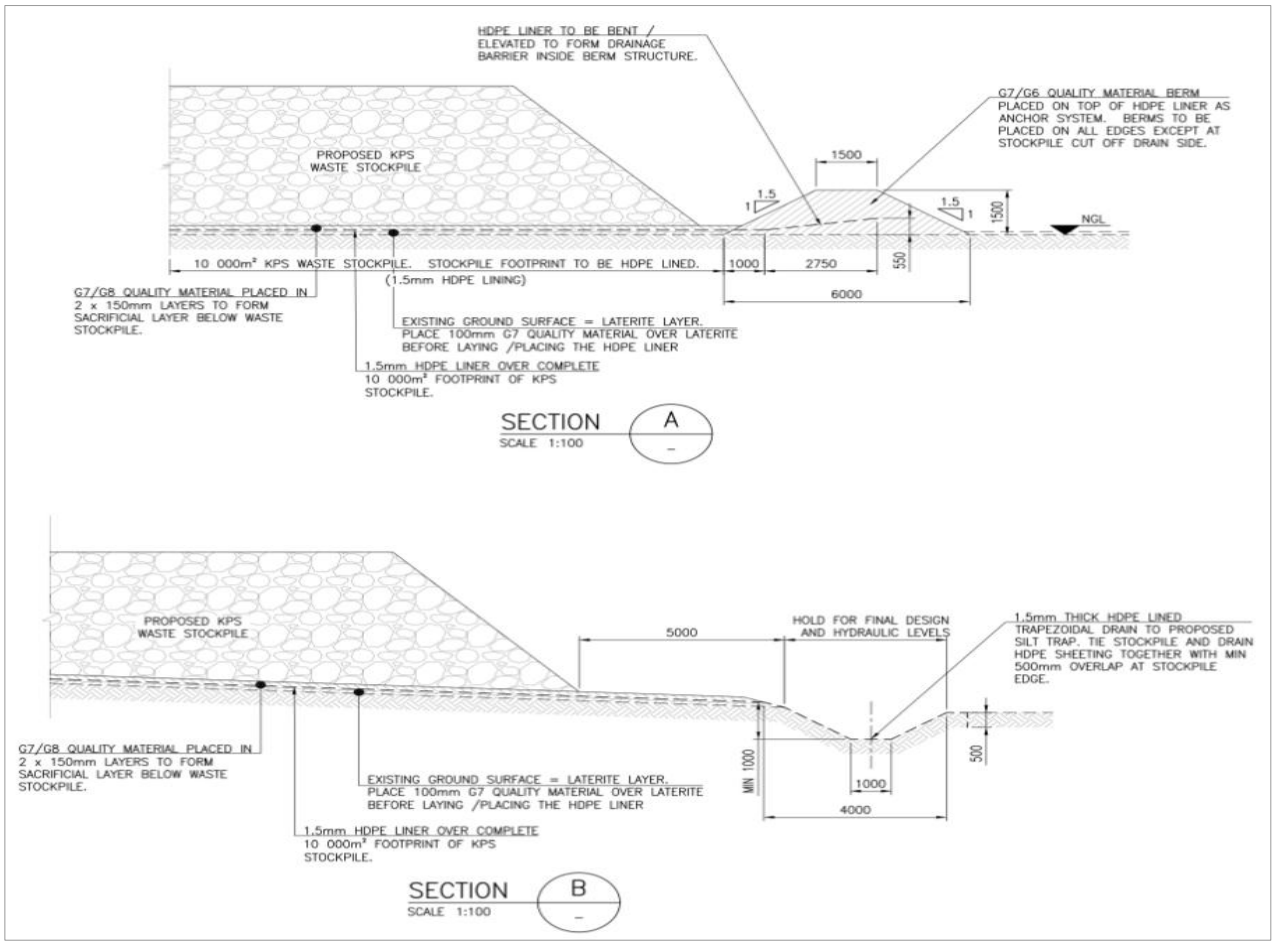
The runoff water from the ROM stockpiles will discharge to the same pollution control facilities, as it is expected to contain high concentrations of copper. The water will be pumped to the operations for processing.

Figure 18.18 Typical KPS Waste Stockpile Facility - Plan View



Source: DRA 2026

Figure 18.19 Typical KPS Lined Stockpile and Contact Drain – Sectional View



Source: DRA 2026

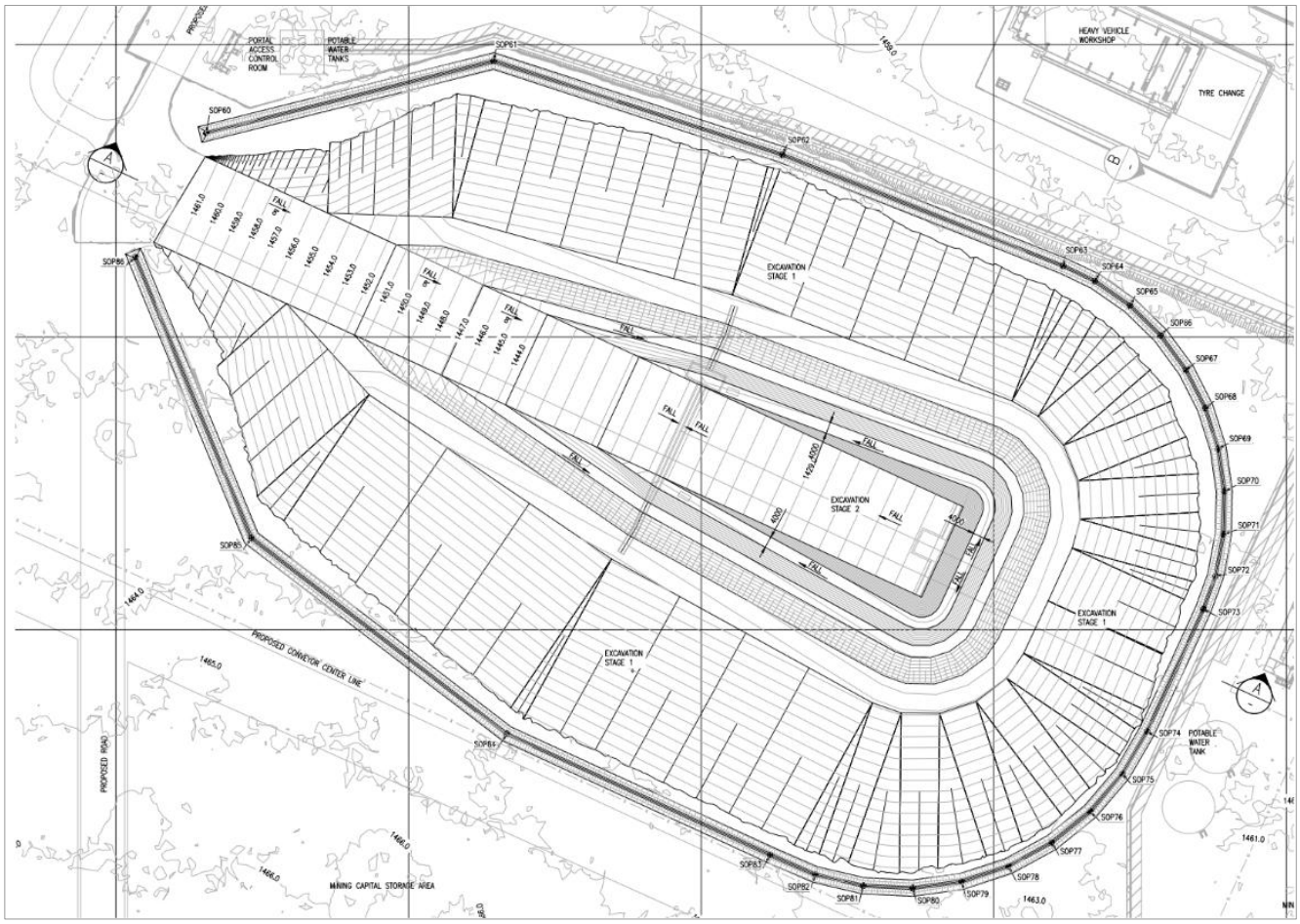
18.3.5 Mine Access and Rock Handling

18.3.5.1 Box cuts

Access to the underground orebodies will be established via portals developed within the highwalls of box cuts, in conjunction with decline shaft systems. No alternative access methodologies were considered as part of this study. Refer to Figure 18.20 and Figure 18.21 for a plan layout and section of a typical box cut.

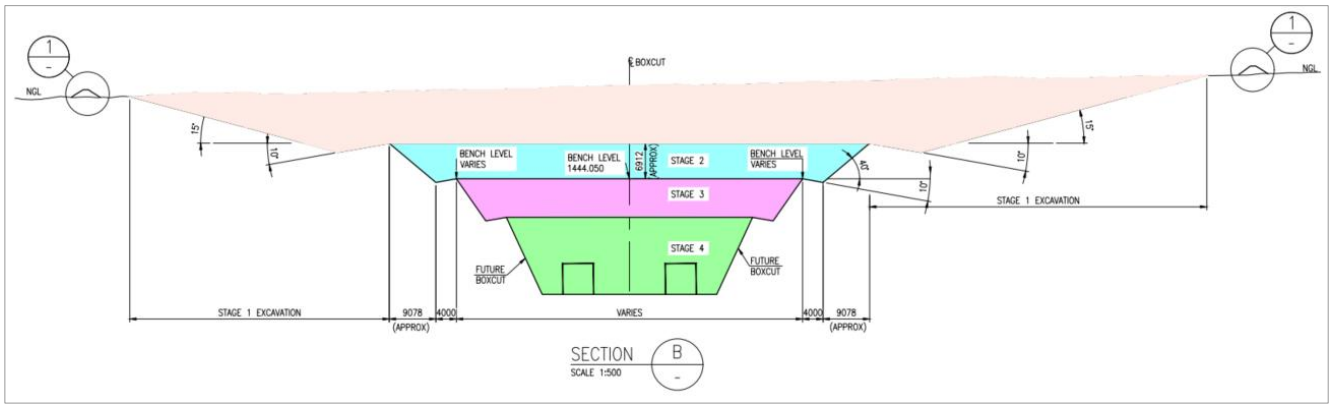
Detailed geotechnical investigations and engineering design for each box cut at each mine access position will be conducted prior to commencement of construction.

Figure 18.20 Typical Box cut Plan Layout



Source: DRA 2026

Figure 18.21 Typical Section Through Box cut



Source: DRA 2026

18.3.5.2 Rock Handling

Waste rock from underground mine, off-reef development, will be stockpiled on Waste Rock Dumps allocated for each mine. Due to the proximity of the existing Kakula North and Kamoa 1 conveyor decline shafts to the respective processing facilities, ore is transferred directly to the concentrator plants through overland conveyors. Ore from the rest of the underground mines (including ore from Kakula South Decline Shaft) is stockpiled separately for re-loading into haul road-going trucks and transportation to Bulk Reclaim Tips at the front-end of the existing Kakula and Kamoa 1 Concentrator plants.

18.3.6 Services and Utilities

18.3.6.1 Dewatering

Facilities for the handling and treatment of mine return water pumped from underground comprise shaft collars for dewatering borehole piping, overland buried pipelines and settling dams as detailed below.

18.3.6.2 Dewatering Shaft Collar

The dewatering shaft column and collar will include all the terracing, civil collar and pipe supports.

18.3.6.3 Buried Pipelines

The buried pipelines include the clearing and stripping of the pipeline servitude, excavations, supply and installation of piping, fittings and instrumentation and the backfilling of trenches.

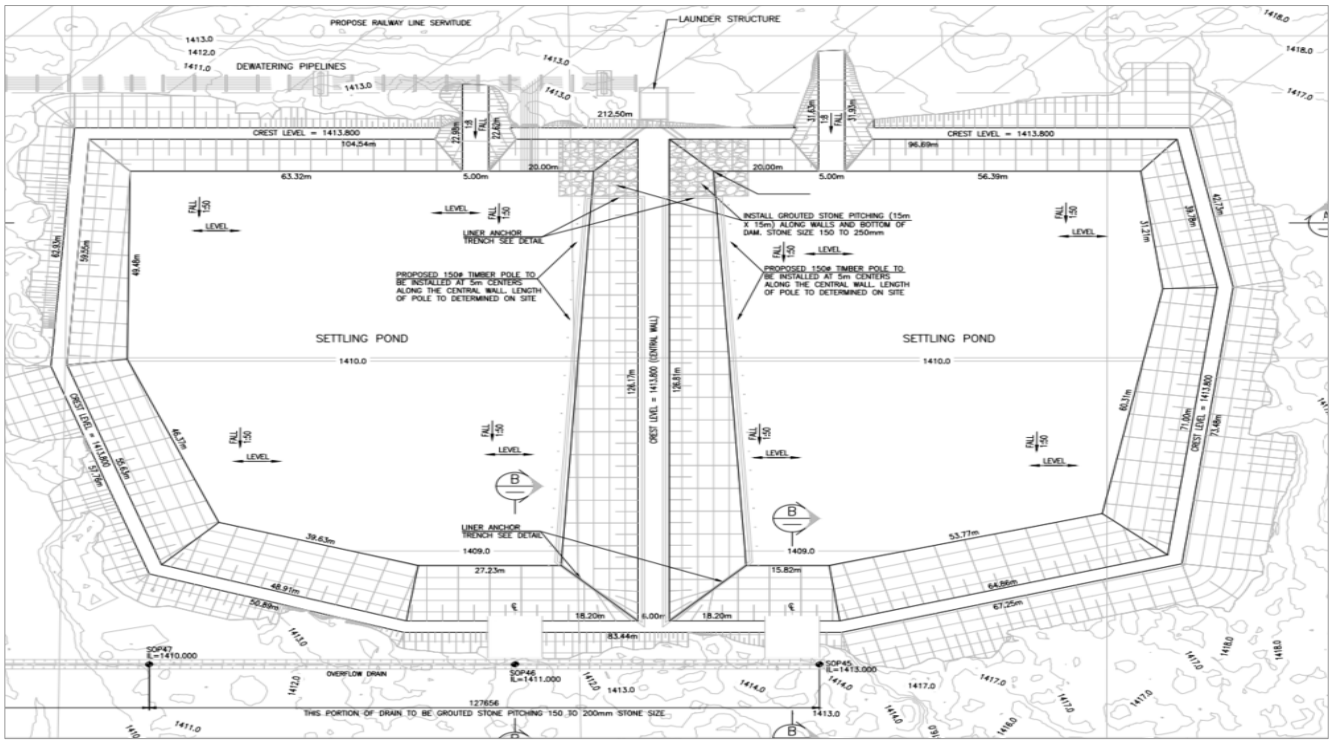
18.3.6.4 Surface Settlers

Three typical surface settling pond designs were developed and utilized for costing purposes in the MRMR Update. These designs were based on varying discharge capacities of 900 l/s, 1200 l/s, and 1600 l/s, respectively. Refer to Figure 18.22, Figure 18.23 and Figure 18.24 for the typical plan views for each of these settling dams. The settlers will consist of two lined earth ponds, one operational, one standby. This allows for the maintenance and cleaning of the one cell whilst the other is in operation.

To reduce overall project costs and improve the efficiency of the multi-stage dam system, including the vertical transfer dams, the number of underground transfer dams were reduced where possible. This was achieved by increasing the hydraulic capacity of the surface settling dams, thereby allowing a greater volume of flow to be managed above ground.

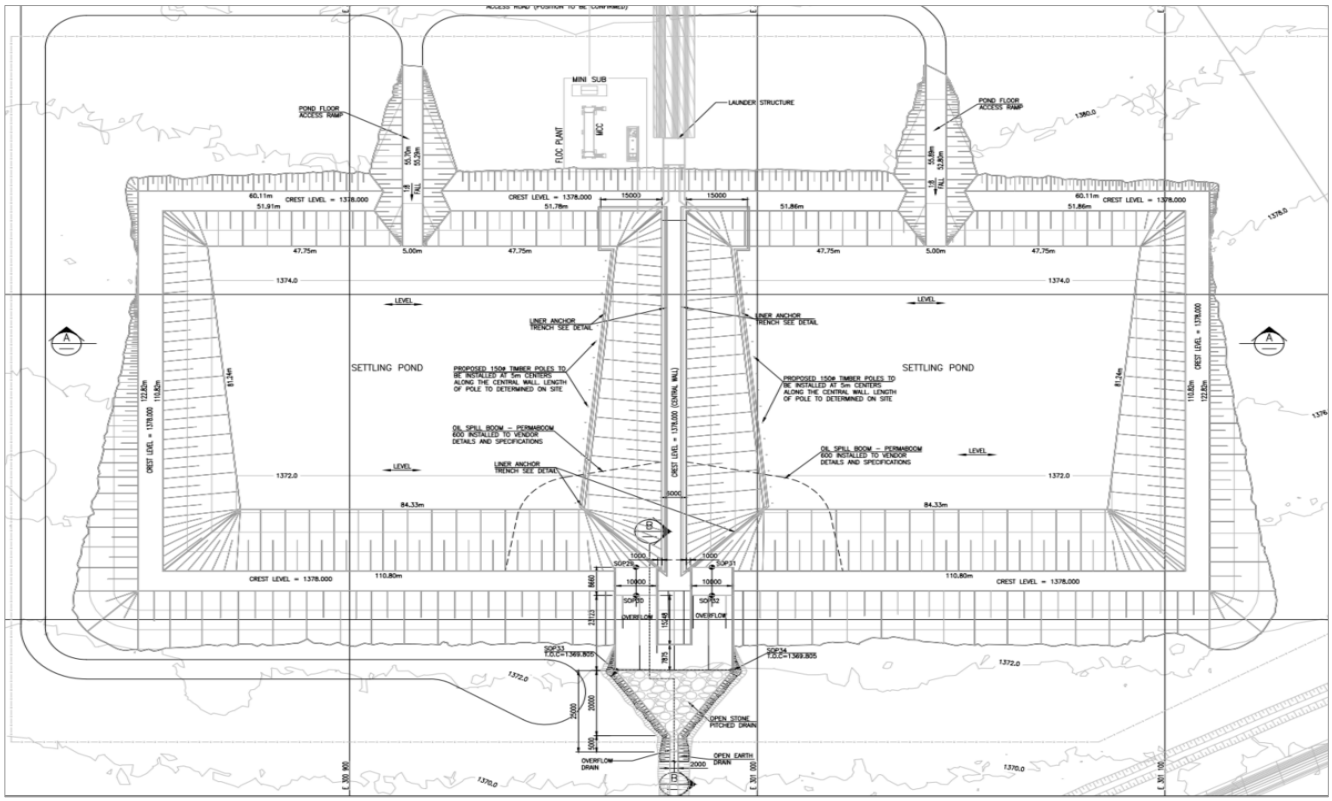
As part of this optimization, the design capacity of certain settling ponds was increased to accommodate an increased discharge from 1,600 l/s to 2,000 l/s. Although this specific pond was not explicitly modelled at the higher capacity, the associated quantities were proportionally factored to establish a representative cost estimate for the larger pond.

Figure 18.22 Typical 1600l/s Settling Pond Arrangement - Plan View



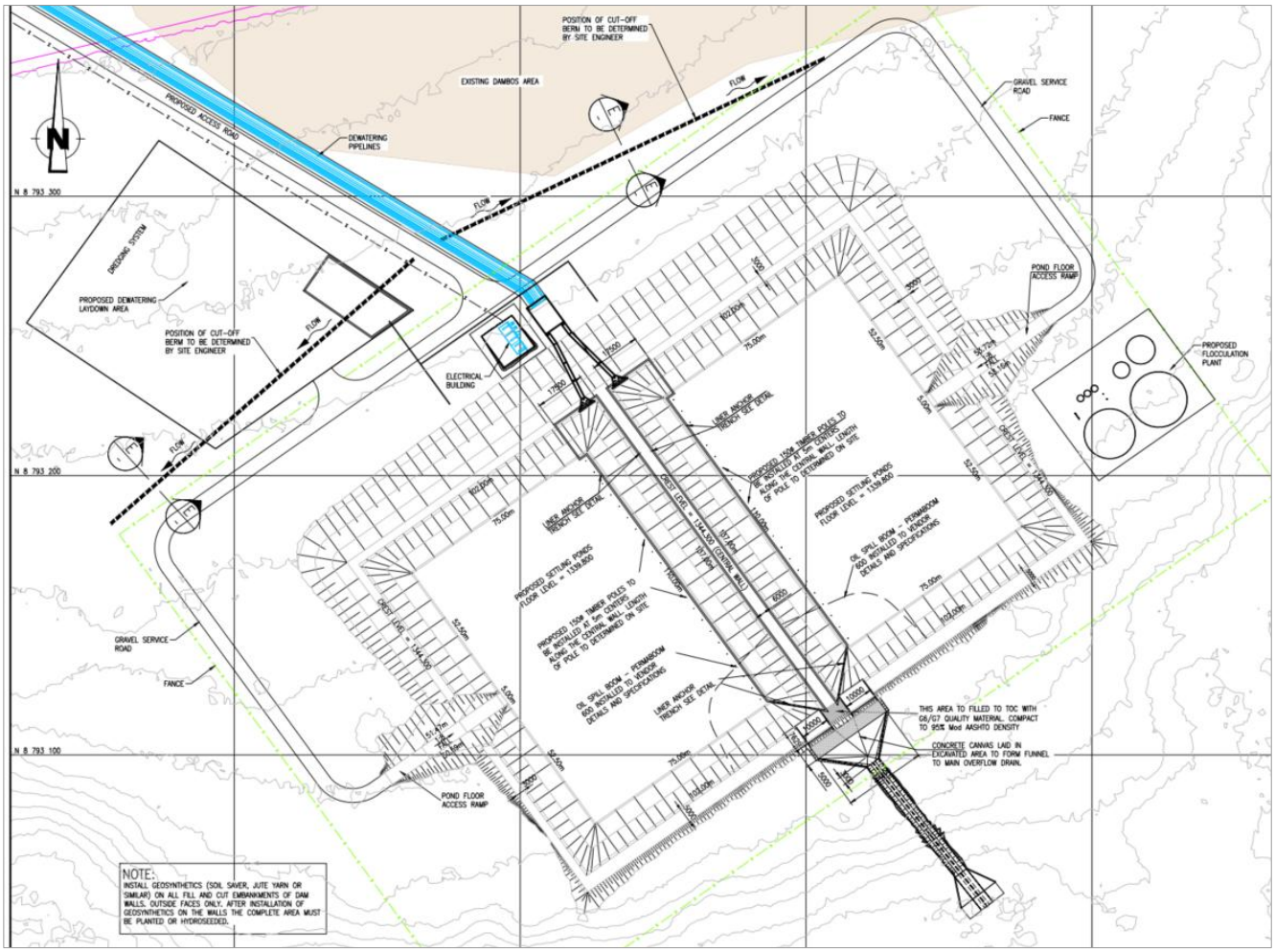
Source: DRA 2026

Figure 18.23 Typical 1200l/s Settling Pond Arrangement - Plan View



Source: DRA 2026

Figure 18.24 Typical 900l/s Settling Pond Arrangement - Plan View



Source: DRA 2026

18.3.6.5 Compressed Air Supply

Compressed air for underground refuge chambers is supplied from compressor houses on surface through borehole piping. The design for each compressor house includes three air cooled, oil-free compressors, two operating and one on standby, an air receiver, overland and borehole piping. Ancillary equipment including manual and automatic control valves, pressure relief valves, silencers and pressure relief valves will be installed in the compressed air piping system to regulate air supply pressure and volume to the refuge chambers.

18.3.6.6 Potable Water

Potable water demand is currently supplied from treated borehole water. There are two existing equipped wellfields, located at the Kamoa 1 and Kakula mines. The existing wellfields are not fully equipped, however, and further equipping will occur as the potable water demand increases.

An additional wellfield is planned near the proposed Kamoa 4 Mine, the wellfield will be located within the West Scarp fault zone.

An updated water-balance is required to establish the anticipated future potable water demand.

18.3.6.7 Fire Water

Firewater supply systems comprising dedicated water storage tanks and pump houses will be provided for all box cut infrastructure and reticulated to various end-user points.

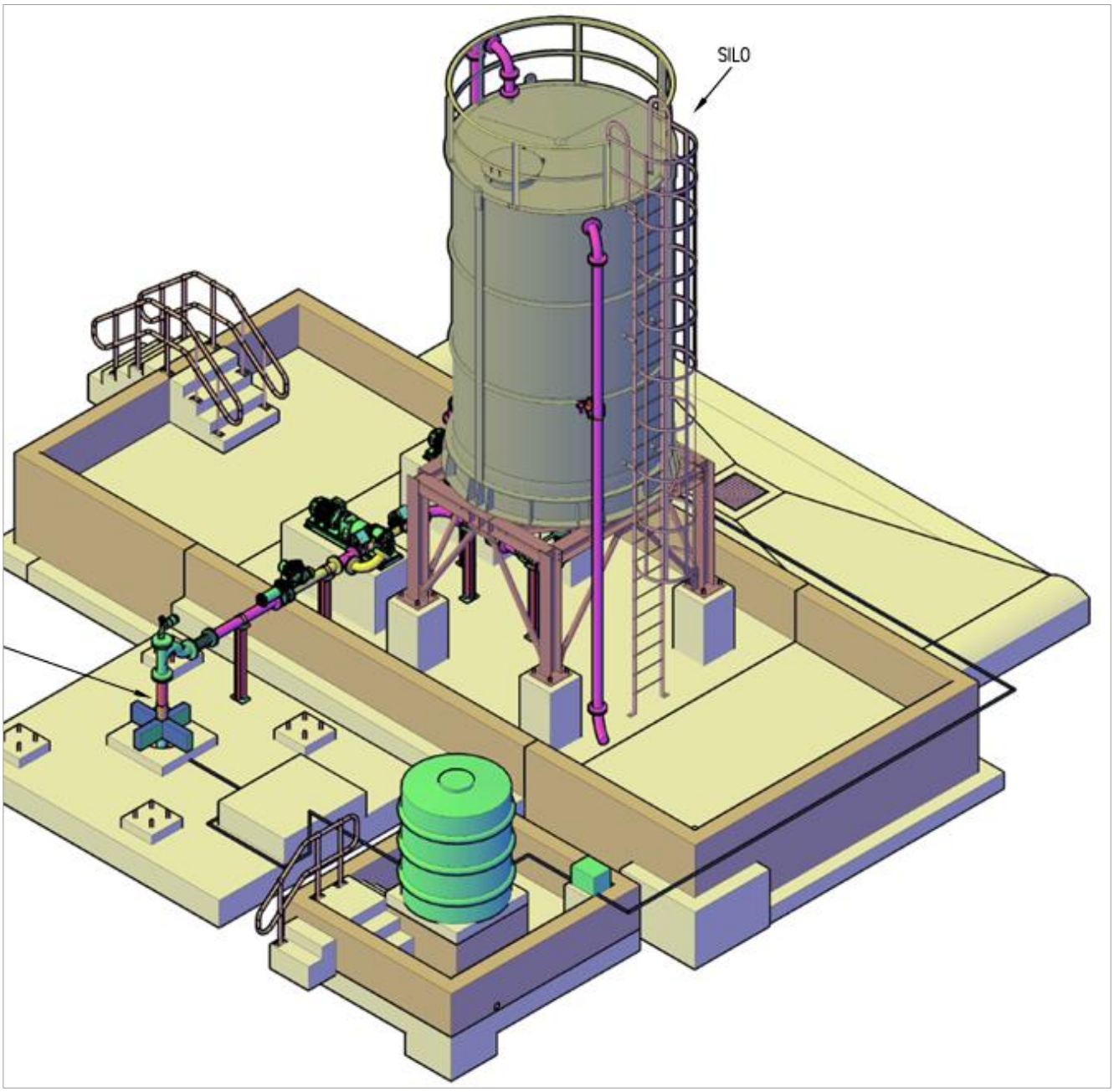
18.3.6.8 Service Water

Service water for surface and underground mining and engineering operations is supplied from recirculated, filtered mine return water. All proposed new mines are anticipated to be water positive and will adopt similar recirculation strategies to limit freshwater intake.

18.3.6.9 Emulsion

Vertical drop emulsion delivery system will be used to deliver non-detonable emulsion and sensitizer from surface to underground bulk storage tanks. Surface facilities include either emulsion silos, batching tanks or receiving kettles, transfer pumps, piping (surface and borehole piping), valves and control equipment. The scope for surface equipment differs across the mines based on the orebody size and planned mining production. Figure 18.25 illustrates surface facilities for a relatively high production mine requiring an emulsion storage silo. Smaller mines will only incorporate receiving kettles and transfer pumps for direct transfer from a surface utility vehicle to borehole pipes.

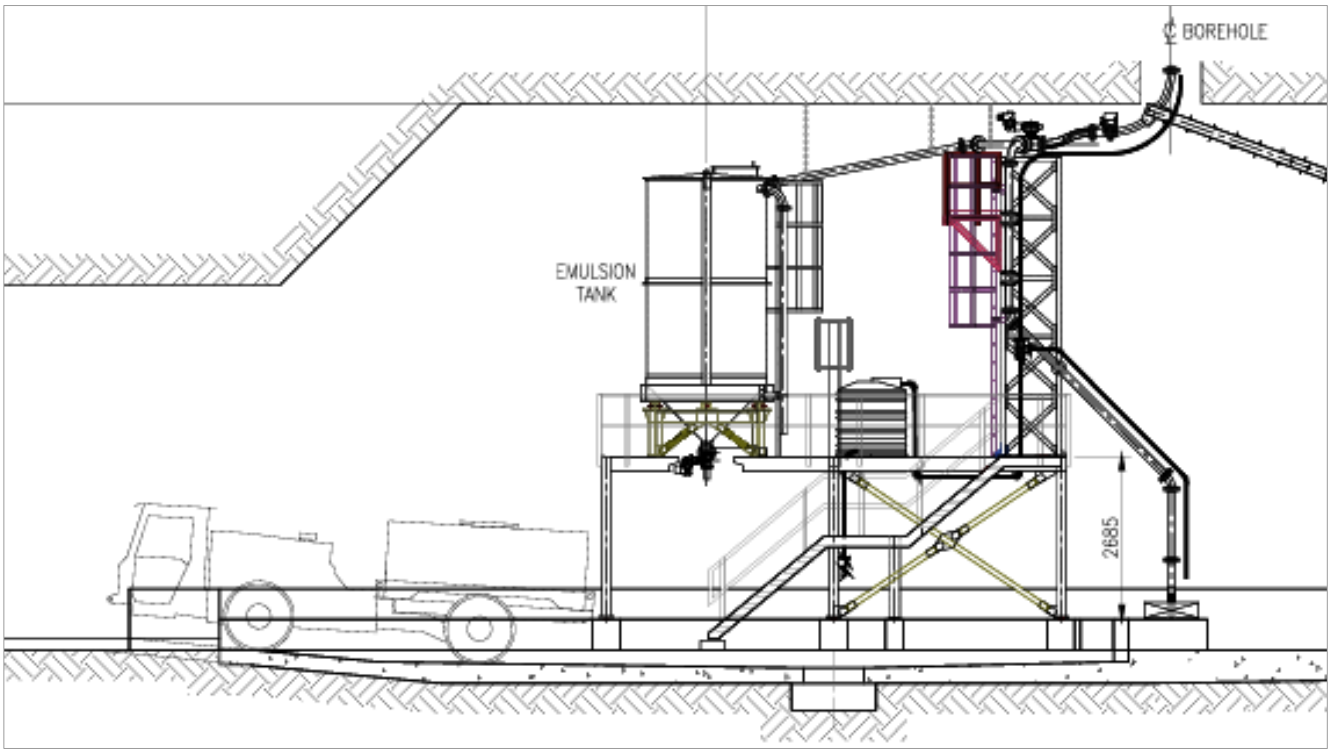
Figure 18.25 Surface Emulsion and Sensitizer Storage



Source: DRA 2026

Underground receiving facilities for the vertical drop system are located adjacent to main underground workshops and include storage silos and piping and control valves for loading underground charging units as illustrated in Figure 18.26.

Figure 18.26 Underground Emulsion and Sensitizer Plant



Source: DRA 2025

18.3.6.10 Fuel and Lubricants

Diesel fuel and lubricants (mainly engine oil, transmission oil, hydraulic oil and engine coolant) will be supplied to underground bulk storage tanks, located adjacent to main workshops, through vertical drop systems. These systems consist of batch tanks located on surface for measuring and feeding exact volumes to underground storage tanks, suitably rated pipework installed in boreholes, terminating at bulk storage tanks and ancillary equipment for controlled gravity feed, pipe emptying and accounting of product volumes. Fuel and lubricant batching, storage and dispensing facilities are of specialized design and will be supplied as specialist turnkey packages.

18.3.6.11 Shotcrete

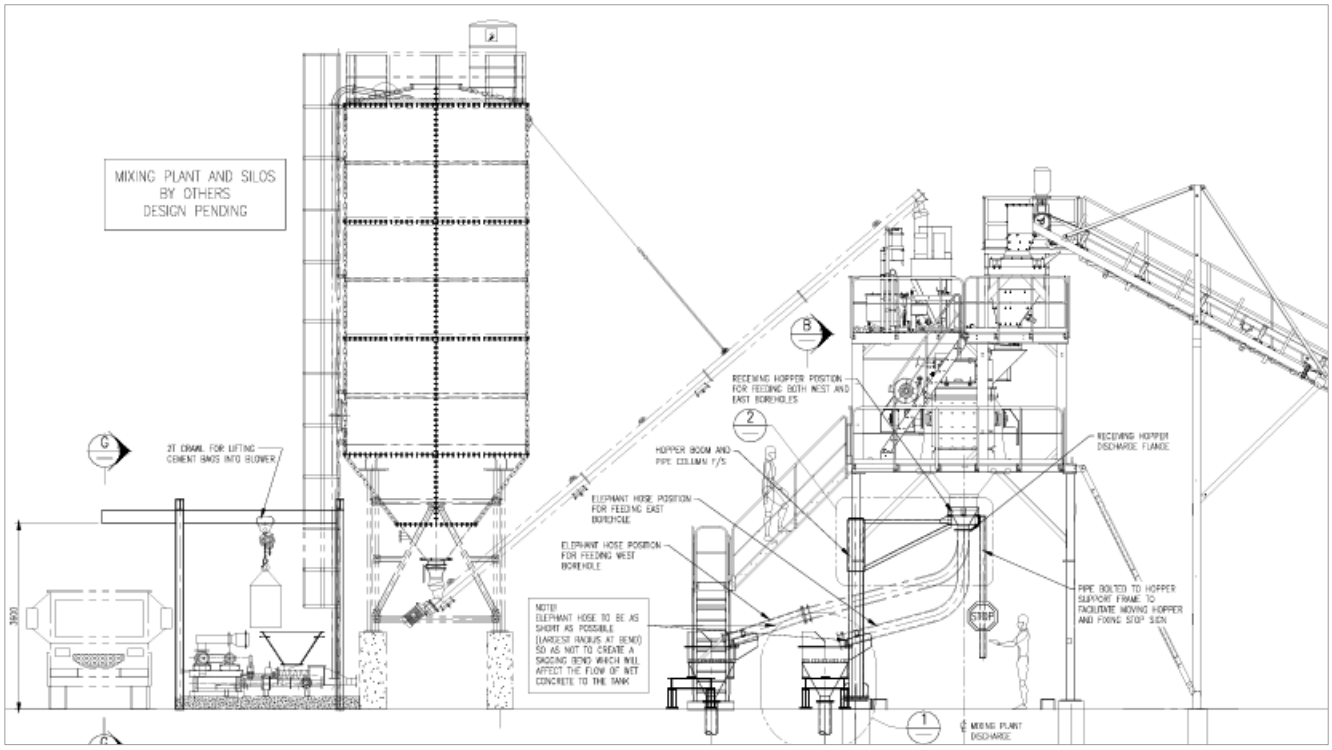
A shotcrete batching plant will be installed at each mine's box cut infrastructure terrace to supply the mine's development and production shotcrete requirements. Shotcrete plants consist of:

- Cement storage silo, complete with cement bulk bag loading facilities and screw discharge feeder.
- Aggregate loading hopper complete with aggregate discharge conveyor belt.
- Water supply and dispensers.
- A ring-pan mixer with hydraulic operated segment gates.
- Weighing equipment
- Ancillary equipment including instrumentation, control and electrical equipment.

Batch plants are specialized installations supplied as turnkey packages. Wet shotcrete mix from the batch plant is discharged directly into vertical boreholes for feeding underground shotcrete handling plant

discussed in Section 18.8.3.7, or into road-going concrete mixer trucks for discharge into remote located vertical drop systems. These vertical drop systems will consist of receiving tanks or hoppers for batch feeding the wet shotcrete to the underground handing plant. Figure 18.27 shows a fixed batch plant (excluding aggregate hoppers) with wet shotcrete discharged directly into borehole slicklines.

Figure 18.27 Surface Shotcrete Batching Plant



Source: DRA 2025

Water storage tanks will be provided for flushing mixers and borehole slicklines to prevent shotcrete from setting and causing blockages.

18.3.6.12 Site Wide Services

18.3.6.13 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 utilizes 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 potable water purposes.

18.3.6.14 Buried Services

All buried services are designed according to South African National Standards, SANS 1200. These include:

- Earthworks, i.e. trenching (SANS 1200D)
- Bedding for pipes (SANS 1200LB)
- Piping, valves and valve chambers, anchor/thrust blocks and manholes (SANS 1200L)

- Concrete and miscellaneous metal work (SANS 1200G).

18.3.6.15 Sewage Treatment and Reticulation

The mine's domestic sewage requirements enable collection from various facility points for routing through 100–250 Nominal Bore (NB) pipelines. 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 administration 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 sewage treatment 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 sterilization zone, and the sludge will be sent back to the anaerobic section for further processing. In the sterilization section, the processed water will be polished and sterilized using plasma ozone. From the sterilization section the (clear liquid) treated effluent water is discharged.

18.3.6.16 Weighbridge

A weighbridge will be installed at the main entrance/exit of each mine.

18.3.6.17 TSF Pipeline Servitude

The tailings delivery and return pipeline servitude is an extension of the existing servitudes from the Kamoa and Kakula concentrators. The proposed servitudes are required to extend the existing servitudes to the proposed TSF Sites.

A 50 m servitude will be bush cleared for the entire route of the pipelines and a laterite service road will be constructed. Four HDPE tailings pipelines will be routed along the tailing pipeline servitude to the TSF to transport tailings material from the process plant(s) to the TSF. Three more HDPE lines will be installed in the same servitude for TSF return water, back to the process plant(s). The pipes will be placed unsupported on the ground. The powerline to the TSF will be installed in the same servitude.

18.3.6.18 Airports

Not Applicable

18.3.7 Buildings and Workshops

The building allocations were aligned with the Major or Minor infrastructure allocations. The exception to this allocation is Kamoa 2 East Box cut, that will utilize Kamoa 1 Mining infrastructure with minimal buildings allocations.

Six types of buildings will be used for the Kamoa-Kakula surface infrastructure:

- **Containerized**: Limited use of containerized buildings for early works applications only. These buildings are shipping containers, converted to a building for a specific use.
- **Prefabricated:** Limited use of prefabricated buildings for early works applications only. These buildings are erected on a concrete slab. A galvanized 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 1,2 mm gauge, high-strength galvanized 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 to blockwork buildings. 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 prefabricated modular E-houses that are delivered to site and placed on in-situ cast civil bases with steel platforms for access.

Table 18.8 and Table 18.9 list the various buildings included in the Kamoa-Kakula MRMR Update. This includes all the modular buildings and sheeted steel structures with civil bases. The allocations are based on the Major and Minor infrastructure split for the new mines. Table 18.10 shows the allocated electrical buildings, including mine infrastructure upgrades.

Table 18.8 Proposed Buildings for the Major Infrastructure Mines

Item No.	Description	Floor Area (m^2)	Kamoa 3 Mine	Kamoa 4 Mine *	Kakula West Mine
1	TM3 Workshop	1 174	Yes	Yes	Yes
2	HVW	2 516	Yes	Yes	Yes
3	HVW Canopy Extension	1 392	Yes	Yes	Yes
4	Mine Stores	4 636	Yes	Yes	Yes
5	Cement Stores (New)	298	Yes	Yes	Yes
6	Oil Stores	225	Yes	Yes	Yes
7	Gas Stores	225	Yes	Yes	Yes
8	Executive Offices	707	Yes	Yes	Yes
9	Portal Access Control	100	Yes	Yes	Yes
10	Mine Gatehouse 3 Red Zone	73	Yes	Yes	Yes
11	Mine Lamp Room	2 216	Yes	Yes	Yes
12	Main Admin Office	1 625	Yes	Yes	Yes
13	Mine Office Building 1	418	Yes	Yes	Yes
14	Main gatehouse	154	Yes	Yes	Yes

Item No.	Description	Floor Area (m^2)	Kamoa 3 Mine	Kamoa 4 Mine *	Kakula West Mine
15	Central Receiving stores	1 454	Yes	Yes	Yes
16	Capital Laydown Area	2 241	Yes	Yes	Yes
17	Gatehouse Bus Stop	73	Yes	Yes	Yes
18	Weighbridge Office	173	Yes	Yes	Yes
19	Proto Building	924	Yes	Yes	Yes
20	**Control Room / Data Centre**	**562**	**Yes**	**Yes**	**Yes**

Note: * Applicable to Kamoa 4 West Mine

Table 18.9 Proposed Buildings for the Minor Infrastructure Mines

Item No.	Description	Floor Area (m^2)	Kansoko SUD Mine	Kamoa 2 Mine*[1]	Kamoa 4 Mine *[2]	Kamoa 5 Mine	Kamoa 6 Mine
1	TM3 Workshop	880.5	Yes	Yes	Yes	Yes	Yes
2	HVW	754.9	Yes	Yes	Yes	Yes	Yes
3	HVW Canopy Extension	417.6	Yes	Yes	Yes	Yes	Yes
4	Mine Stores	1 390.8	Yes	Yes	Yes	Yes	Yes
5	Cement Stores (New)	149.0	Yes	Yes	Yes	Yes	Yes
6	Oil Stores	56.3	Yes	Yes	Yes	Yes	Yes
7	Gas Stores	56.3	Yes	Yes	Yes	Yes	Yes
8	Construction Offices	665.5	Yes	Yes	Yes	Yes	Yes
9	Portal Access Control	100.0	Yes	Yes	Yes	Yes	Yes
10	Mine Gatehouse 3 Red Zone	72.5	Yes	Yes	Yes	Yes	Yes
11	Mine Lamp Room	1 108.0	Yes	Yes	Yes	Yes	Yes
12	Main Admin Office	325.0	Yes	Yes	Yes	Yes	Yes
13	Mine Office Building 1	418.4	Yes	Yes	Yes	Yes	Yes
14	Main gatehouse	153.6	Yes	Yes	Yes	Yes	Yes
15	Central Receiving stores	727.0	Yes	Yes	Yes	Yes	Yes
16	Capital Laydown Area	1 120.5	Yes	Yes	Yes	Yes	Yes
17	Gatehouse Bus Stop	72.5	Yes	Yes	Yes	Yes	Yes
18	Weighbridge Office	173.0	Yes	Yes	Yes	Yes	Yes
19	Proto Building	462.0	Yes	Yes	Yes	Yes	Yes
20	Control Room / Data Centre	168.6	Yes	Yes	Yes	Yes	Yes

Note: [1] Kamoa 2 West Mine has only has a portal Access Control Building and Main Gate House (Zone3).
[2] Applicable to Kamoa 4 East Mine.

Table 18.10 Electrical Buildings

Area	Description	Floor Area (m^2)	Kakula Mine [Qty]	Kamoa 1 Mine	Kansko SUD Mne	Kamoa 2 Mine	Kamoa 3 Mine	Kamoa 4 Mine	Kakula West Mine	Kamoa 5 Mine	Kamoa 6 Mine
1	33/11kV Outdoor substation – Multistage or Vertical Transfer Pumpstations	554	4	5	1	3	4	1	8	1	1
2	80 MVA Transformer Extension	422	-	1	-	-	-	-	-	-	-
3	220kV Yard Extension	422	-	1	-	-	-	-	-	-	-
4	33kV Extension - Kamoa Mines KMCMS	554	-	1	-	-	-	-	-	-	-
5	**33/11kV Outdoor substation – Box cuts**	**554**	**-**	**-**	**1**	**2**	**1**	**3**	**3**	**1**	**1**

18.3.7.1 Trackless Mining Machinery Workshops

Trackless mining machinery workshops will be constructed at the box cut infrastructure terraces for all new mines. The workshops are for routine maintenance services, breakdown repairs and major overhauls for primary and secondary mining fleets. Surface workshops will comprise service pits and bays, a 15-ton overhead travelling crane spanning the full workshop length and includes offices and other amenities.

The following auxiliary facilities are provided to compliment main workshop operational and mining production requirements:

- New and used lubricants storage, dispensing and pumping equipment.
- Fuel storage and dispensing equipment.
- Tyre storage, handling and maintenance shop.
- Hydraulic and pneumatic shop.
- Compressed air supply and workshop power tools.

A wash bay is provided adjacent to the workshop for washing of machines before service. Oil contaminated water emanating from the workshop and wash bay is channeled to a silt trap and oil separator before clean effluent is discharged to the environment. An oil recovery facility incorporating an oil separator, tanks and pumps is provided for handling recovered oil.

18.3.7.2 Heavy Vehicles Workshop

Maintenance service, repair and major overhauls for tertiary mining fleets which consists of support vehicles including light and heavy-duty vehicles, lifting equipment, buses etc., will be carried out at the heavy-duty workshops located at the box cut terrace of each new mine, adjacent to trackless mining machinery workshop. The workshops will be equipped with service and repair bays, a 5-ton overhead travelling crane spanning the workshop length, lubricant storage and dispensing facilities and other auxiliary equipment for workshop operating requirements.

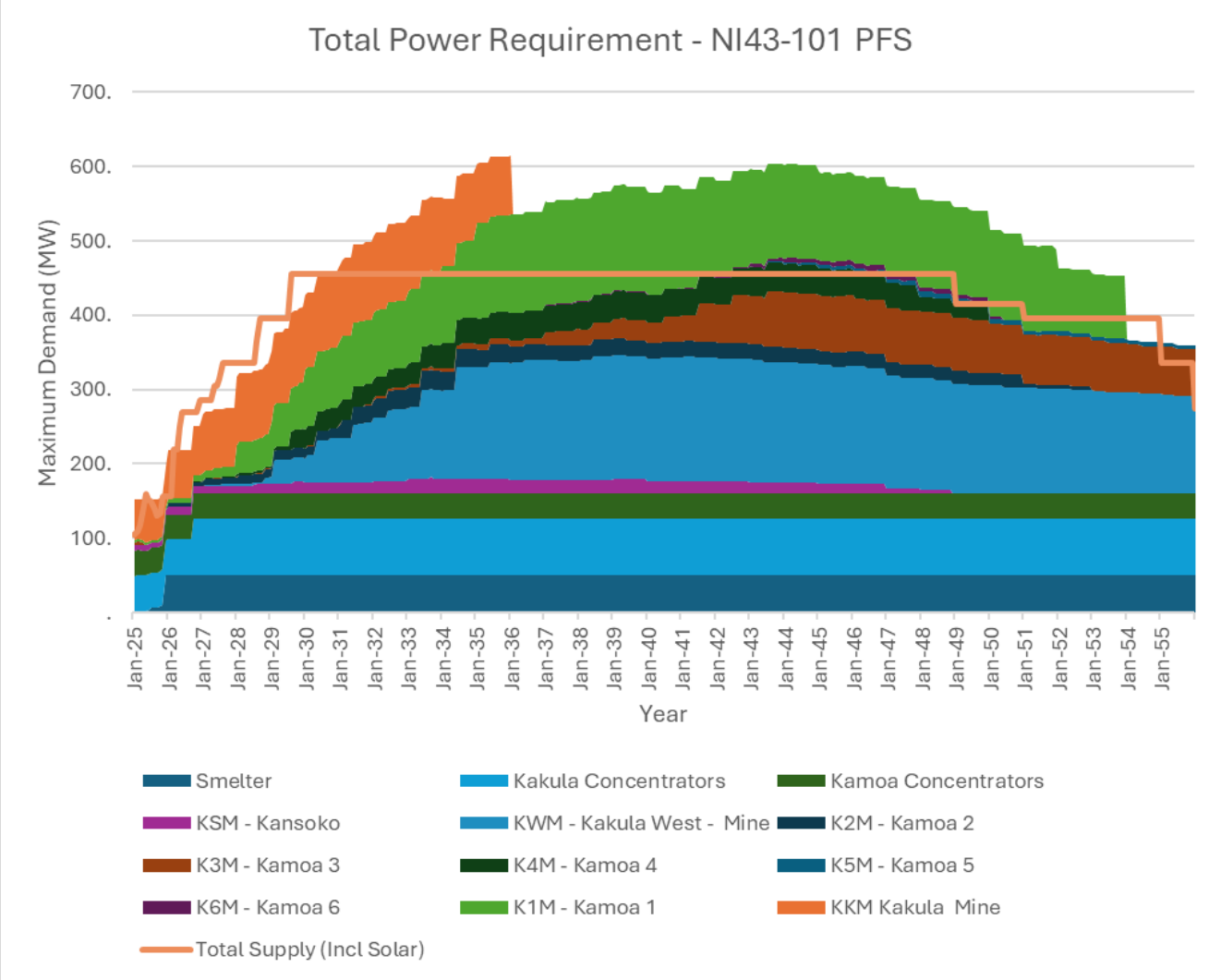
Due to the proximity of these workshops to trackless mining machinery workshops, facilities including wash bays, oil water separator, tyre repair shop and refueling will be shared between the two workshops.

18.4 Power supply

18.4.1 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 as compiled for the Kamoa Life-of-mine NI43-101 PFS was used as load list inputs. The mechanical power requirements were subjected to load capacity de-rating, diversity, and utilization 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 a MD load profile in kW. The Maximum demand profile is illustrated in below Figure 18.28.

Figure 18.28 Maximum Power Demand Kamoa-Kakula Total Power Requirement



Source: DRA, 2026.

The power requirements are as per the latest designs for the following mines, Kakula, Kamoa 1, Kamoa 2, Kansoko and SUD mines, Kamoa 3, Kamoa 4, Kamoa 5, Kamoa 6. Kakula West mines, Concentrator plants and Smelter.

Substation Block Diagrams displaying the power distribution have been developed. These Substation Block Diagrams are included in the Appendixes.

Typical SLDs displaying power distribution have been developed for typical facilities based on existing designs. These SLDs are included in the Appendixes.

18.4.2 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 extended 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. The upgrade of unit 5 (G25) was completed in November 2025 and the available capacity will increase to 125 MW in May 2027. Inga II hydropower plant is shown in Figure 18.29.

Figure 18.29 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 18.30.

Figure 18.30 HVDC OHL from INGA to Lubumbashi

Source: 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 Life-of-Mine project, a total of 620 MW supply will be required. The future power requirements exceed the available capacity and additional power suppliers will have to be identified. The additional power will be sourced from IPPs and running available generator plant.

18.4.2.1 NRO to the Switch Yards to Kamoa 1 and Kakula

Power is currently supplied to Kamoa Copper Mine's 220 kV Kakula Consumer Substation (KCS) from the 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. A total of five 220/33 kV 80 MVA transformers were installed at the Kakula switch yard, two transformers for the Smelter, one transformer for Kakula mine, one transformer for Kakula concentrators. The firth transformer is for N-1 redundancy. Three additional transformers will be required for the additional Kakula mining load and the future Kakula West Mine. Alternatively, the 132 kV OHL should be extended to Kakula West Mine and a 220/33 kV substation build at Kakula West Mine.

Three transformers were installed at Kamoa 220/33 kV Substation were installed at the Kamoa 1 switch yard. The supply will conform the N+1 redundancy on the transmission line and transformers. One additional transformer will be installed to supply the future increase in Kamoa Mine load.

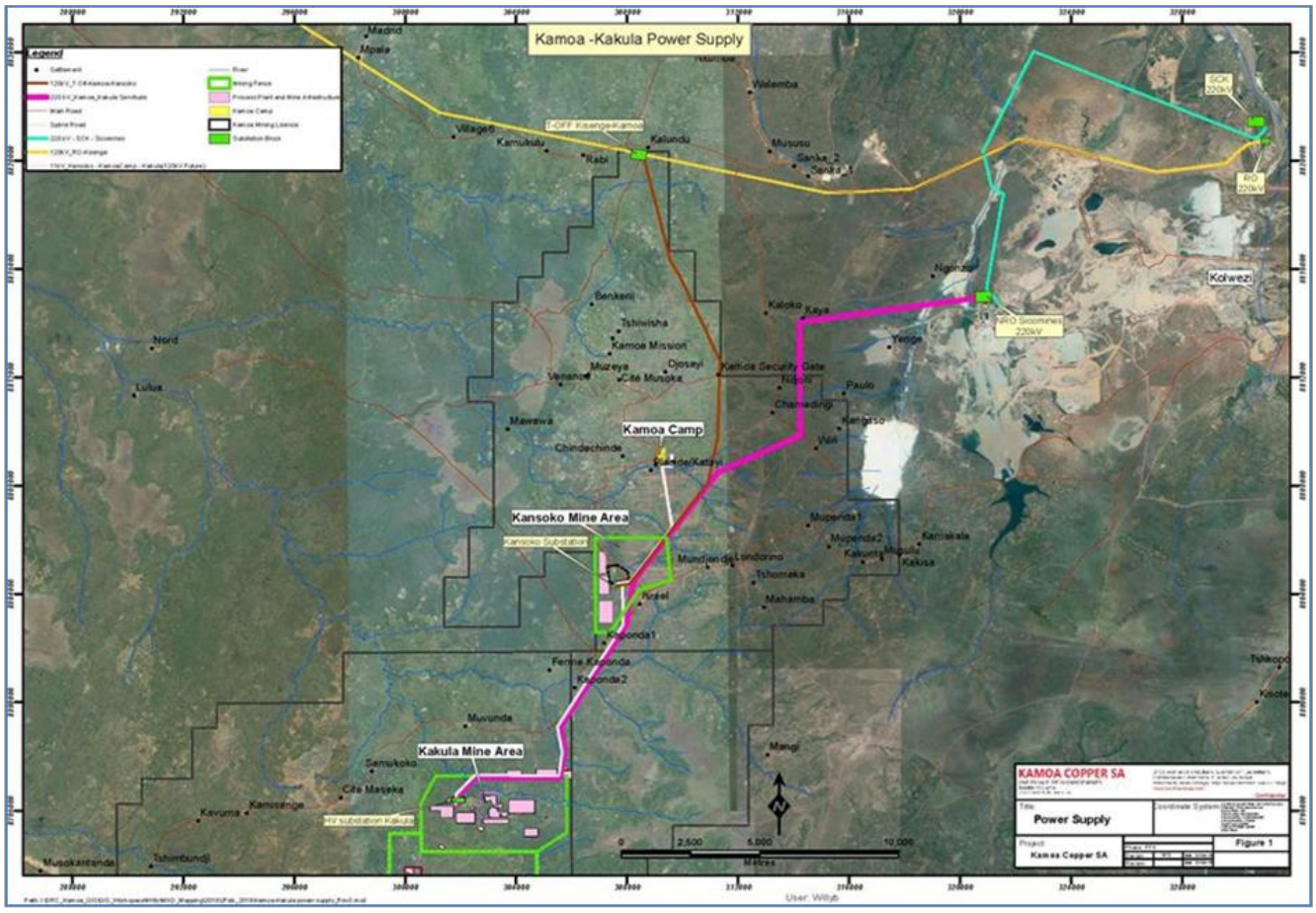
For the future Kamoa 2 mine, Kamoa 3 mine, Kamoa 4 mine, Kamoa 5 mine, Kamoa 6 mine, two additional 80 MVA transformers must be installed at a future Kamoa 220/33 kV substation No.2. This substation will be a Loop-in-Loop-out design in the 220 kV Overhead lines suppling Kamoa/Kakula.

Metering are 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 18.31 indicates the 220 kV line to Kamoa, and the blue line indicates an existing OHL supplying Sicomines mine.

See Figure 18.31 for a simplified representation of the 220 kV reticulation.

Figure 18.31 High-Level Simplified Representation of 220 kV Reticulation



18.4.3 Electrical, Control and Instrumentation Design

18.4.3.1 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.

18.4.3.2 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

18.4.3.3 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. PFC has been allowed for at all the 33 kV Distribution substations. The final position will be determined during the execution phase.

18.4.4 MV Distribution - Kakula

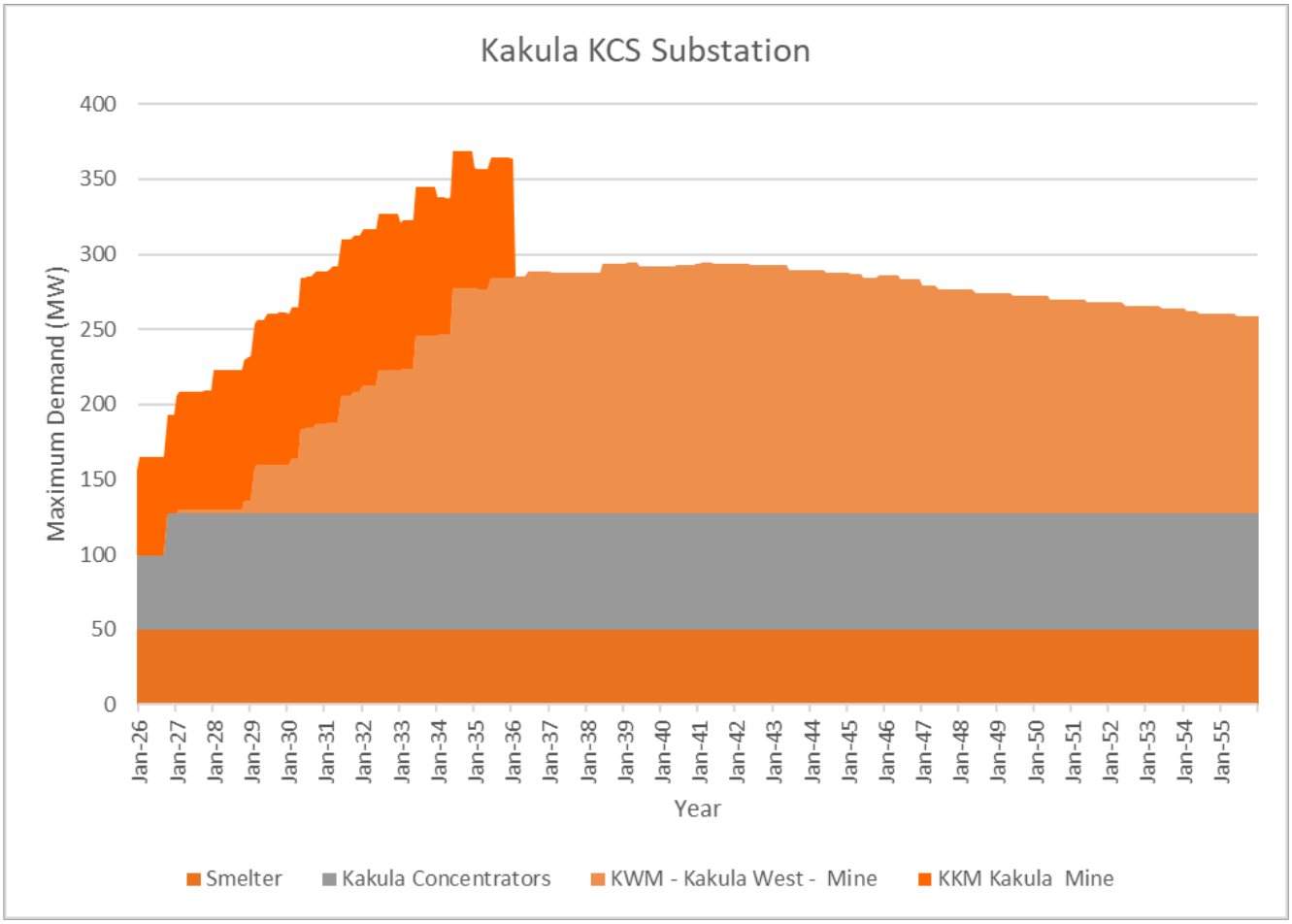
18.4.4.1 220/33 kV Kakula KCS Substation

Bulk 33 kV power for Kakula mine and Kakula Phase1 and Phase2 concentrators are supplied from the 220/33 kV Kakula Consumer substation (Kakula KCS). The substation is equipped with five 80 MVA 132/33 kV transformers. Two transformers supply power to the 33 kV Smelter substation, one transformer for the Kakula Concentrators, one transformer for Kukula mine and the fifth transformer is for redundancy. The power distribution to all MV substations include N+1 redundancy.

The estimate future load at the 220/33 kV Kamoa Consumer substation will require the installation of an additional two 80 MVA 132/33 kV transformer.

Figure 18.32 below indicate the estimated power required at the 220/33 kV Kamoa Consumer (Kamoa KCS) substation.

Figure 18.32 Maximum Power Demand – 220/33 kV Kakula Consumer substation



Source: DRA 2026

18.4.4.2 33 kV Power Distribution – Kakula Mine

Power is distributed at 33 kV to all surface distribution substations and at 11 kV to the underground substations.

Power is distributed to underground Multistage pump stations via a 2 x 30 MVA 33/11 kV Outdoor Substation, a vertical drop cable system and a 11 kV underground substation.

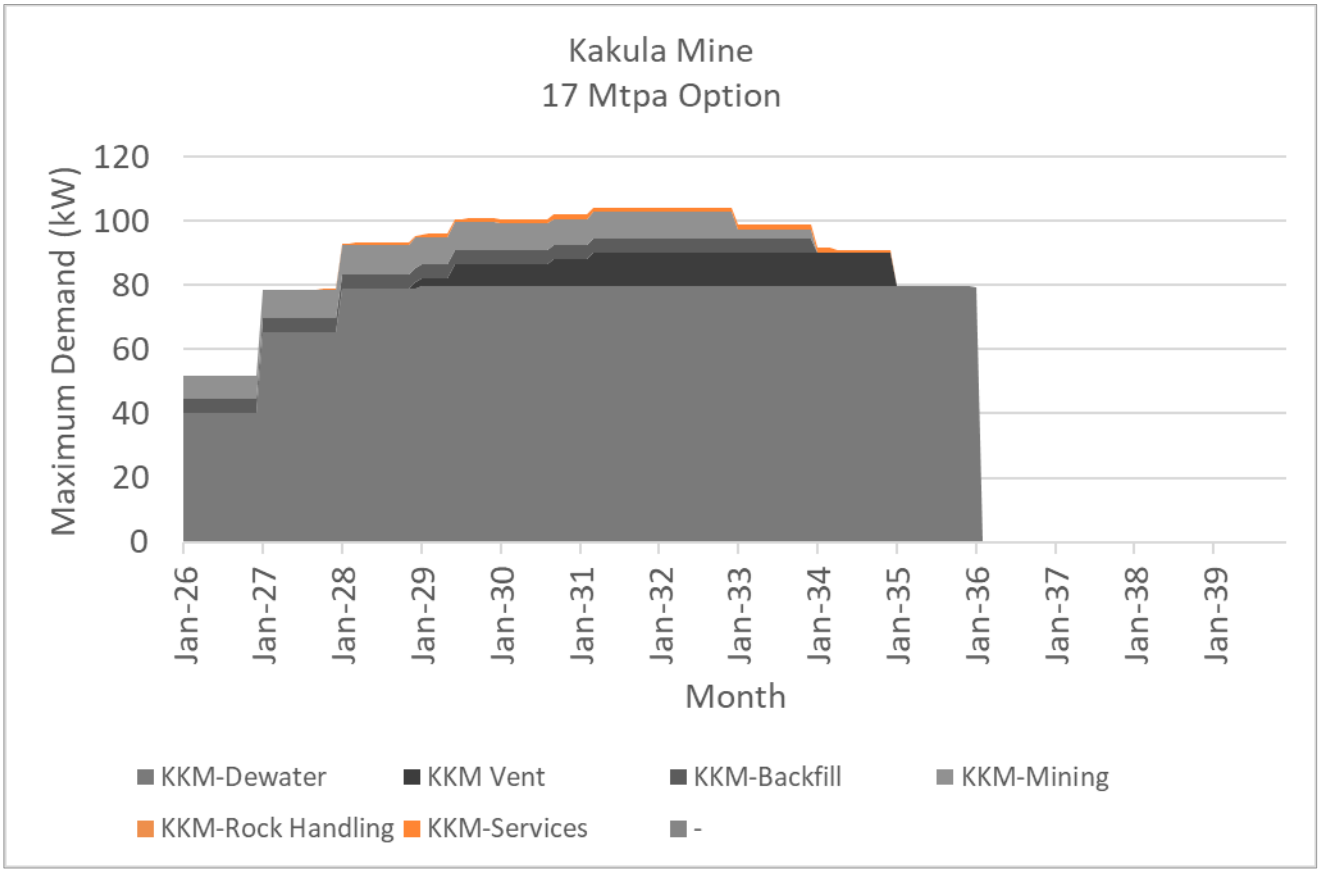
File Name 9843 - KKM SLD excel revA1 display a substation block diagram for the Kakula surface and underground substations. The new substations required will be extensions to the existing Kakula SLDs.

Power is distributed to mining equipment with a combination of 2000kVA 11 kV / 1000 V mini substations, Gulley boxes and local starter panels

Power is distributed to underground infrastructure with a combination of 2500 kVA 11 kV / 690 V transformers and Motor Control Centers (MCCs).

The estimated power required by Kakula Mine is displayed in the Figure 18.33 below. The estimated maximum demand is 133 MW and the largest load is Dewatering.

Figure 18.33 Estimated Maximum Power Demand – Kakula Mine



Source: DRA 2026

18.4.4.3 Power Distribution – Concentrator Plant (Phase 1 and Phase 2)

The Concentrator plant MV distribution is from a dedicated 33 kV Concentrator substation supplied from 33 kV Kakula KCS substation. From this substation there are redundant feeds to the various Concentrator 33 kV substations.

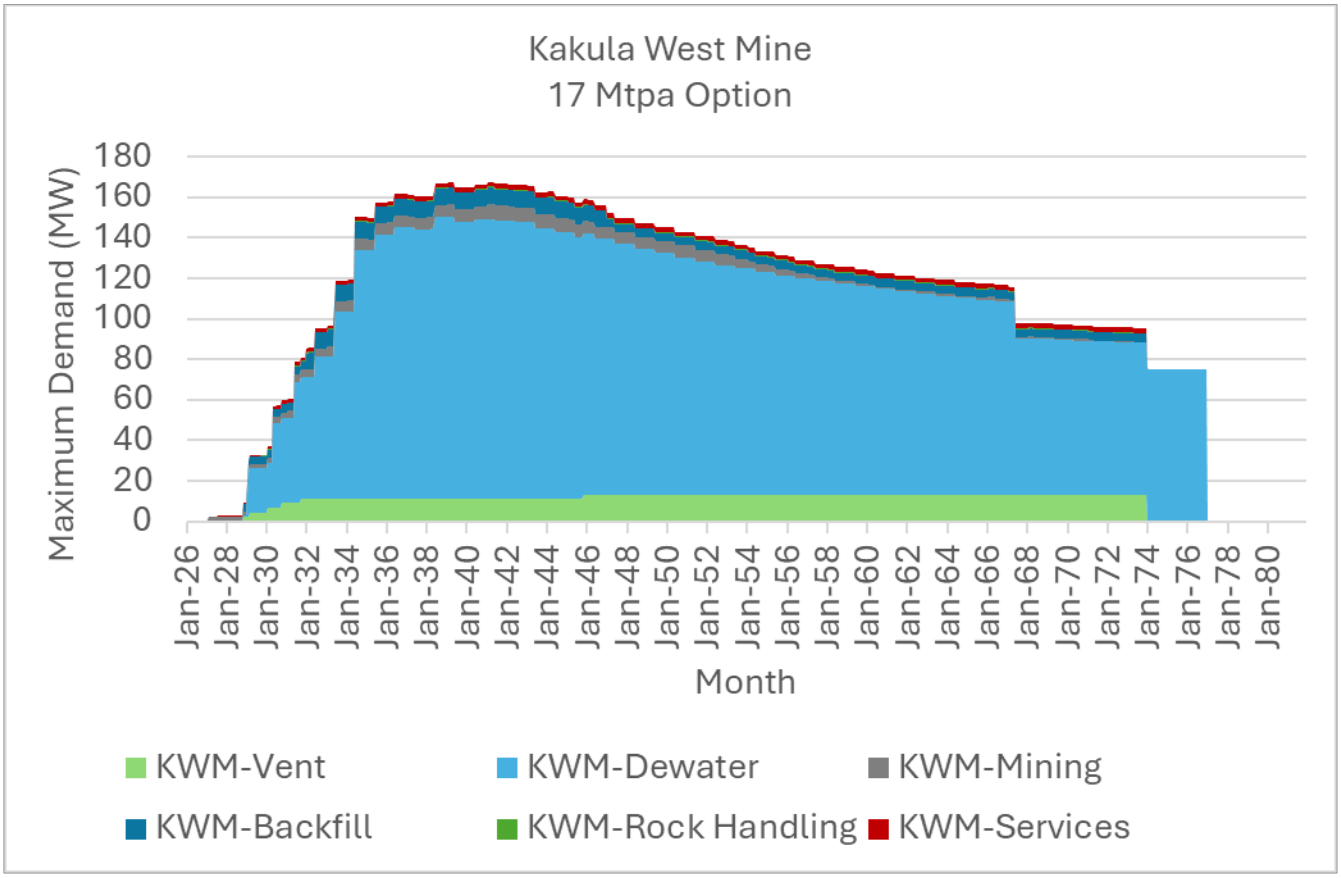
18.4.4.4 33 kV Power Distribution – Kakula West (awaiting new design)

New dedicated 33 kV overhead powerlines will be installed from the 33 kV Kakula KCS substation to supply the two 33 kV Kakula West Outdoor portal substation. This will require the installation of two additional 80 MVA 220/33 kV Transformers at the Kakula KCS. Alternatively, the 220 kV OHLs must be extended from Kakula KCS to a new Kakula West KCS. Power will be distributed from the 33 kV Kakula West portal substations to the various 33kV and 11 kV substations on surface and underground. As

File Name KWM SLD excel rev A display a substation block diagram for the Kakula West Mine surface and underground substations.

The estimated power required by Kakula Mine is displayed in the Figure 18.34 below.

Figure 18.34 Estimated Maximum Power Demand – Kakula West Mine



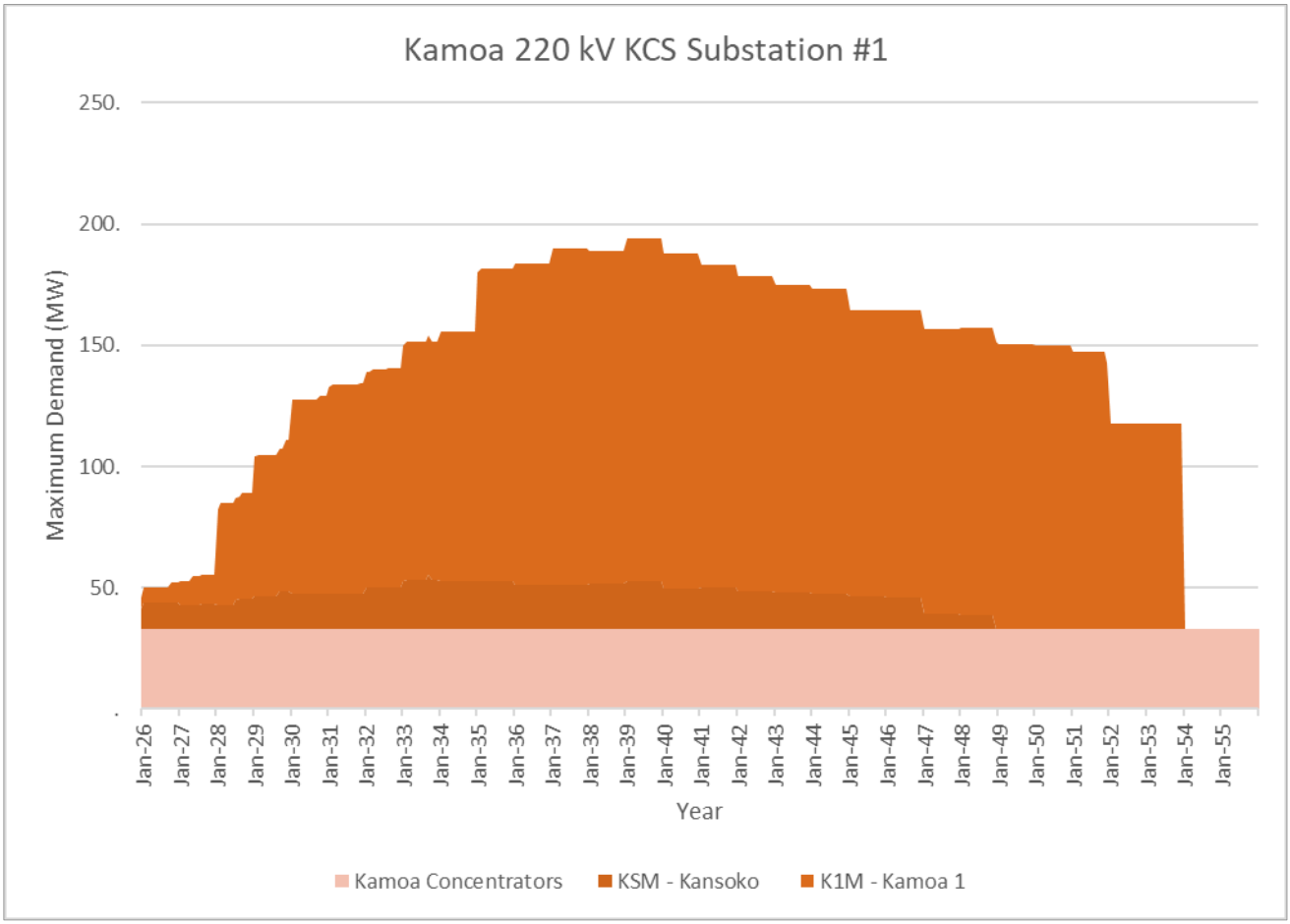
Source: DRA 2026.

18.4.5 220/33 kV Kamoa KCS Substation #1 (Existing)

Infrastructure is supplied from the 220/33 kV Kansoko KCS substation installed at Kansoko (adjacent to the Kamoa Concentrator). The substation is equipped with three 80 MVA 220/33 kV transformers, one transform supplies the Phase 3 concentrator, the second transformer supply the Kamoa Central Mining substation (KCMCS) and the third transformer is for redundancy. Kamoa 1 surface and underground infrastructure is supplied from the KCMCS substation. At present power is distribution from this substation at 33 kV to Kamoa Phase 3 Concentrator, Kamoa 1 Mine and Kansoko Mine.

The estimated power required by Kamoa KCS is displayed in Figure 18.35.

Figure 18.35 Maximum Power Demand – 220/33 kV Kamoa Consumer substation



Source: DRA 2026.

Figure 18.35 display the maximum demand power for the Kamoa KCS #1 substation. The estimated load will require the installation of a fourth 80 MVA transformer to ensure redundancy.

18.4.5.1 Power Distribution – Kamoa Concentrator Plant

The Kamoa Concentrator plant (Phase 3) 33 kV distribution is from a dedicated plant 33 kV MV substation supplied from 33 kV Kamoa KCS substation. From this substation there are redundant feeds to the 33 kV Plant substations via cables on racking.

LV power is distributed via 33 kV / 690 V transformers and MCCs.

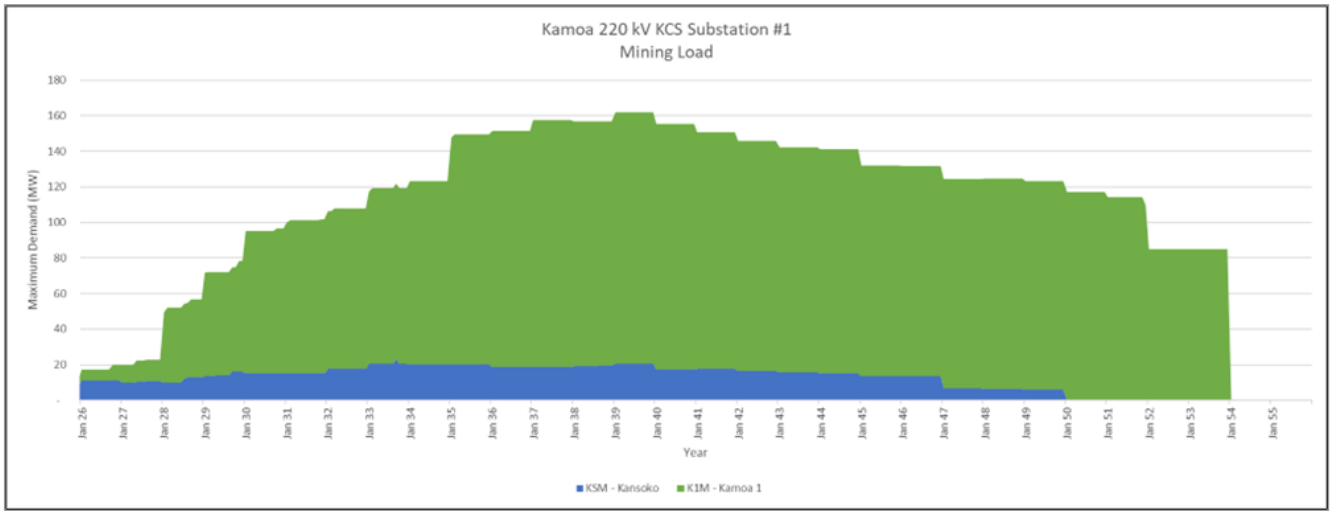
18.4.5.2 33 kV Power Distribution – Kamoa 1

Power is distributed to Kamoa 1 via two 80 MVA rated OHLs to the Kamoa KMCMS. The design allows for an N-1 redundancy to ensure supply reliability. Power is distributed at 33kV to the various Kamoa 1 substations via cables and OHLs.

File Name 9843 - K1M SLD Excel Rev A display a substation block diagram for the Kamoa 1 surface and underground substations. The new substations required will be extensions to the existing Kamoa 1 SLDs.

Figure 18.36 display the maximum demand power for the Kamoa 1 mine. Due to the high dewatering power requirements an additional 80 MVA transformer will be required at KMCMS #1 substation.

Figure 18.36 Maximum Power Demand –Kamoa 1 Mine



Source: DRA, 2026.

Power is distributed to mining equipment with a combination of 2,000 kVA 11 kV / 1,000 V mini substations, gulley boxes and local starter panels.

Power is distributed to underground infrastructure with a combination of 2,500 kVA 11 kV / 690 V transformers and Motor Control Centres (MCCs).

18.4.5.3 33 kV Power Distribution – Kansoko Mine and SUD mine

The Kansoko 33/11 kV substation is supplied from the 33 kV Kamoa KCS substation via MV cable. The design allows for an N-1 redundancy to ensure supply reliability. There is an existing power supply via cables installed in the Kansoko decline. Power is distributed to mining equipment with a combination of 2,000 kVA 11 kV / 1,000 V mini substations, Gulley boxes and local starter panels.

The SUD mine will be supplied from the 33 kV Kamoa Central Mining substation (KCMCS) via a T-off from the OHL supplying the existing TSF facilities. The design allows for an N-1 redundancy to ensure supply reliability.

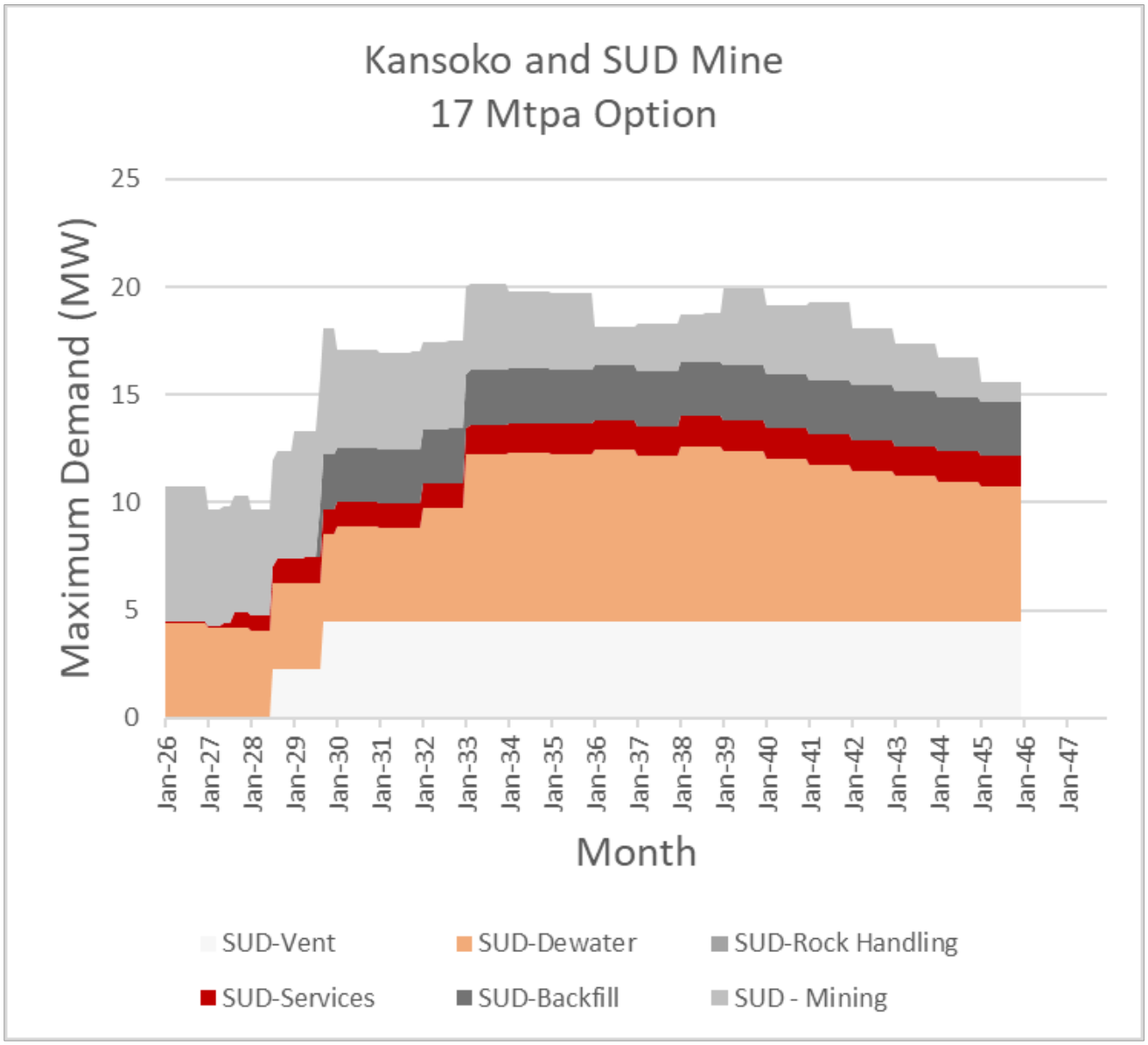
Power is distributed to underground infrastructure via a combination of 2,500 kVA 11 kV / 690 V transformers and MCCs.

File Name KSM SUD excel SLD rev A display a substation block diagram for the Kansoko surface and underground substations and the SUD Surface and underground substations. The new substations required will be extensions to the existing Kansoko SLDs.

Power is distributed to mining equipment with a combination of 2,000 kVA 11 kV / 1,000 V mini substations, Gulley boxes and local starter panels.

Figure 18.37 displays the maximum demand power for the Kansoko and SUD mines.

Figure 18.37 Maximum Demand power for the SUD mine



Source: DRA, 2026.

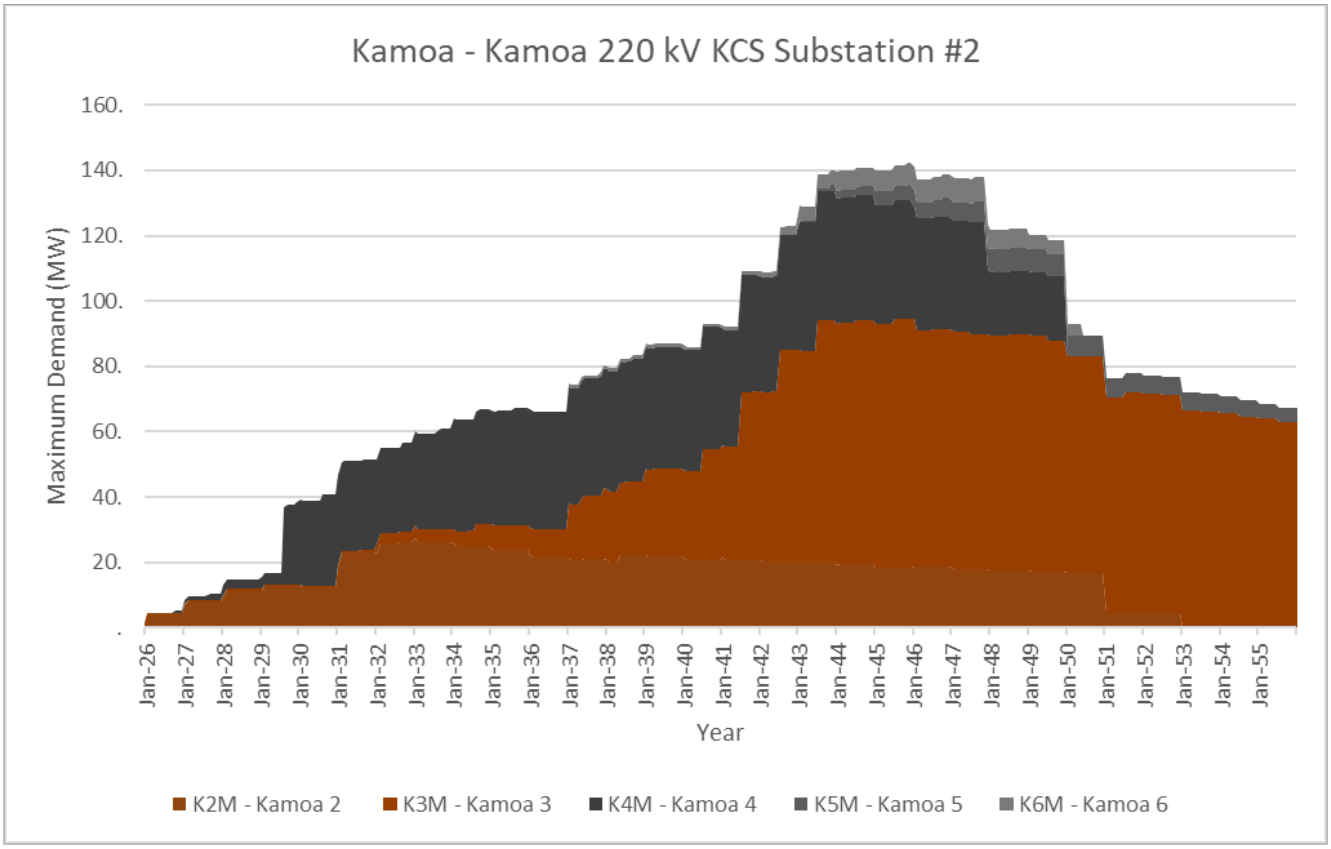
Power is distributed to underground infrastructure with a combination of 2500 kVA 11 kV/690 V transformers and Motor Control Centers (MCCs).

18.4.6 220/33 kV Kamoa KCS Substation #2

Due to the estimated power requirements for the future Kamoa mines a third Snell 220 /33 kV substation will be required. This substation will be supplied via a loop-in-loop-out design from the 220 kV OHLs. This substation will be equipped with three 80 MVA 220/33 kV transformers and will supply power via the third Kamoa KCS Substation #2. Power will be supplied from the Kamoa KCS substation #2 via OHLs to Kamoa 2 Mine, Kamoa 3 Mine, Kamoa 4 Mine, Kamoa 5 Mine and Kamoa 6 Mine.

The estimated load on the Kamoa KCS #2 is displayed in the Figure 18.38 below.

Figure 18.38 Maximum Demand power for Kamoa 220 kV KCS Substation #2



Source: DRA, 2026.

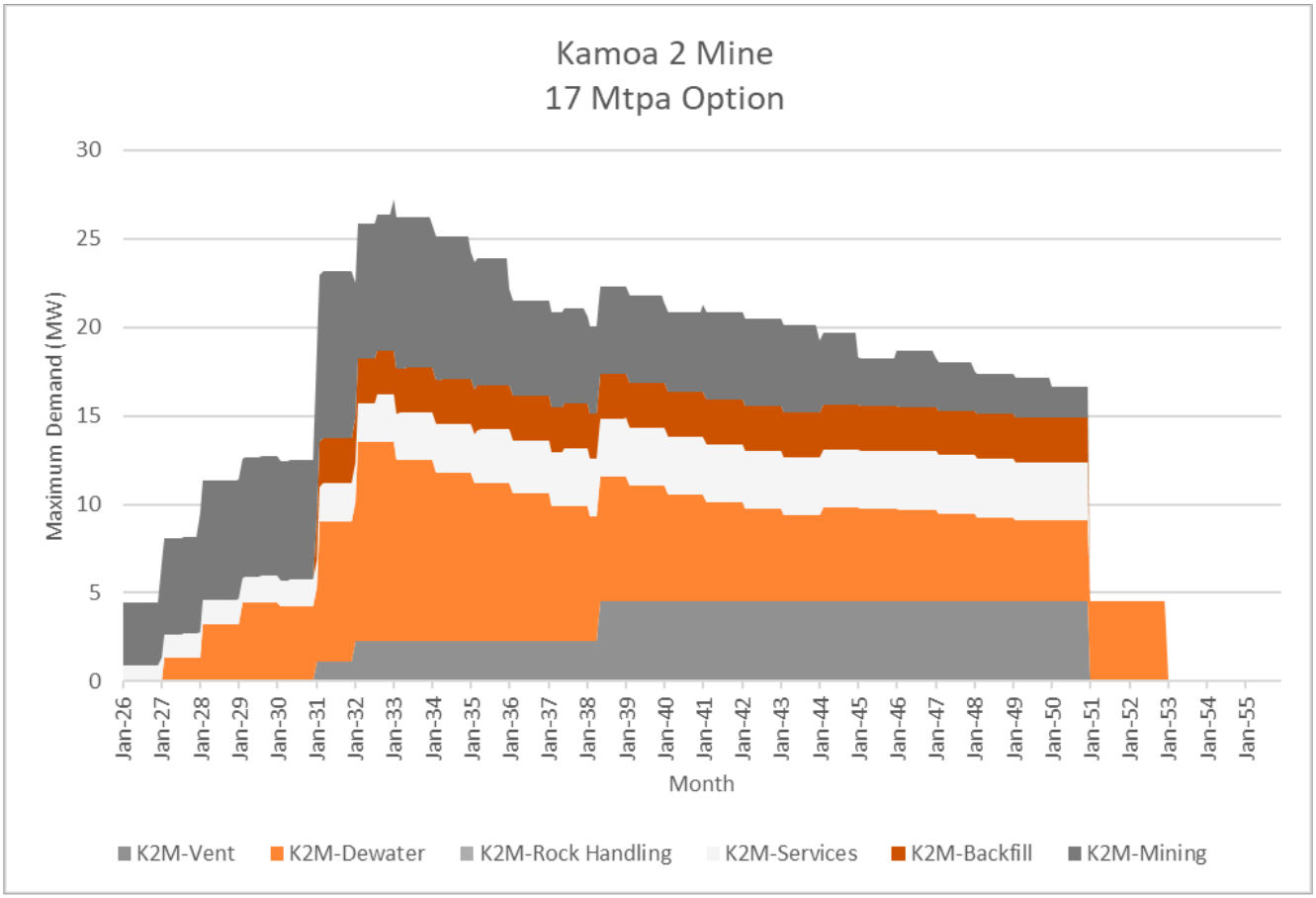
18.4.6.1 33 kV Power Distribution – Kamoa 2

Power will be distributed to Kamoa 2 mine substations via 33 kV OHLs from the 33 kV Kamoa KCS substation #2. The design allows for an N-1 redundancy to ensure supply reliability.

File Name K2M excel SLD rev A display a substation block diagram for the Kamoa 2 surface and underground substations.

Figure 18.39 indicates Kamoa 2 mine power requirements.

Figure 18.39 Maximum Demand power for the Kamoa 2 mine



Source: DRA, 2026.

Power is distributed to mining equipment with a combination of 2,000 kVA 11 kV / 1,000 V mini substations, gulley boxes and local starter panels.

Power is distributed to underground infrastructure with a combination of 2,500 kVA 11 kV / 690 V transformers and MCCs.

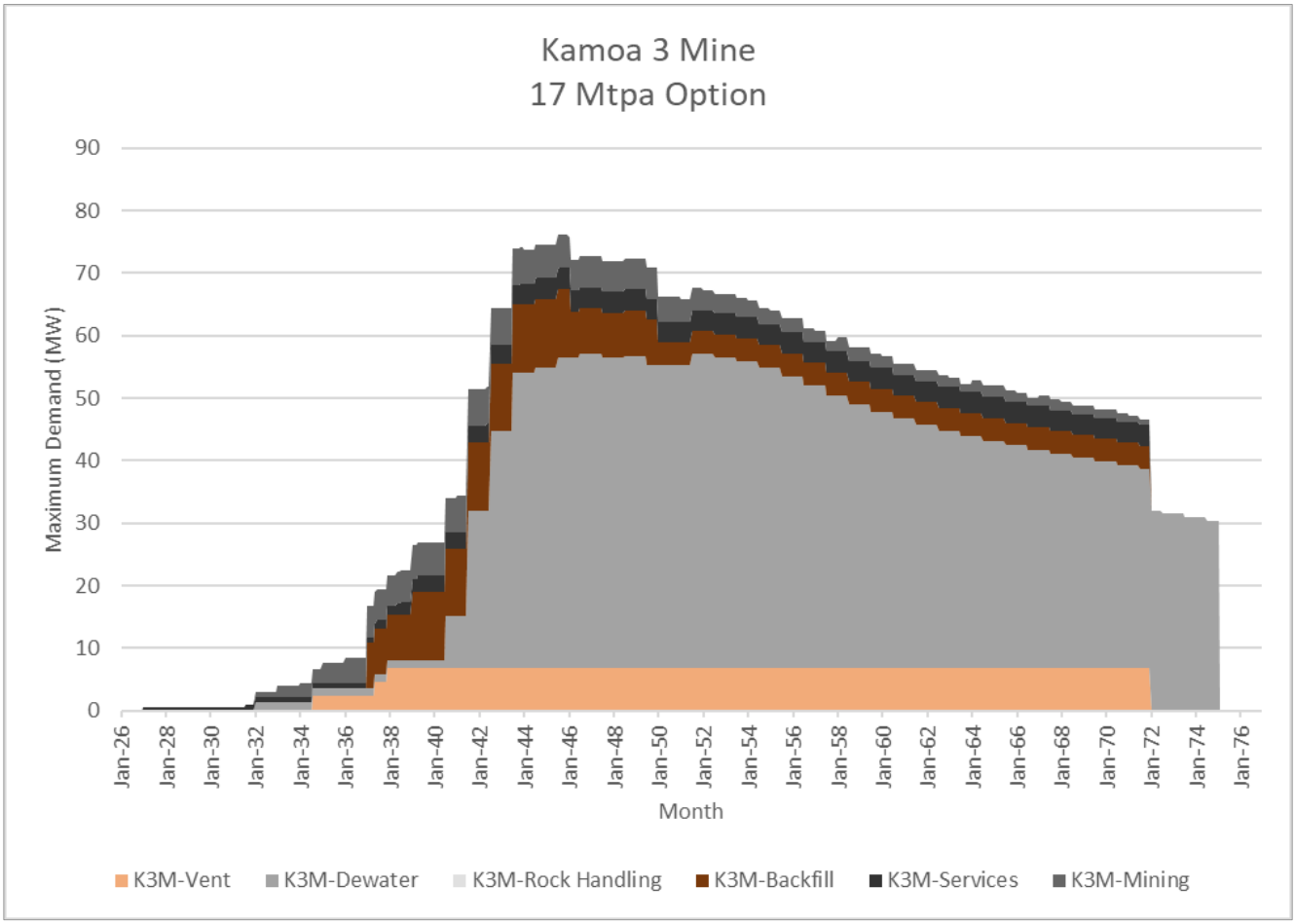
18.4.6.2 33 kV Power Distribution – Kamoa 3

Power will be distributed to Kamoa 3 Mine substations via 33 kV OHLs from the 33 kV Kamoa KCS substation #2. The design allows for an N-1 redundancy to ensure supply reliability.

File Name K3M excel rev A display a substation block diagram for the Kamoa 3 Mine surface and underground substations.

Figure 18.41 indicates the Kamoa 3 mine power requirements.

Figure 18.40 Maximum Demand power for the Kamoa 3 Mine



Source: DRA 2026.

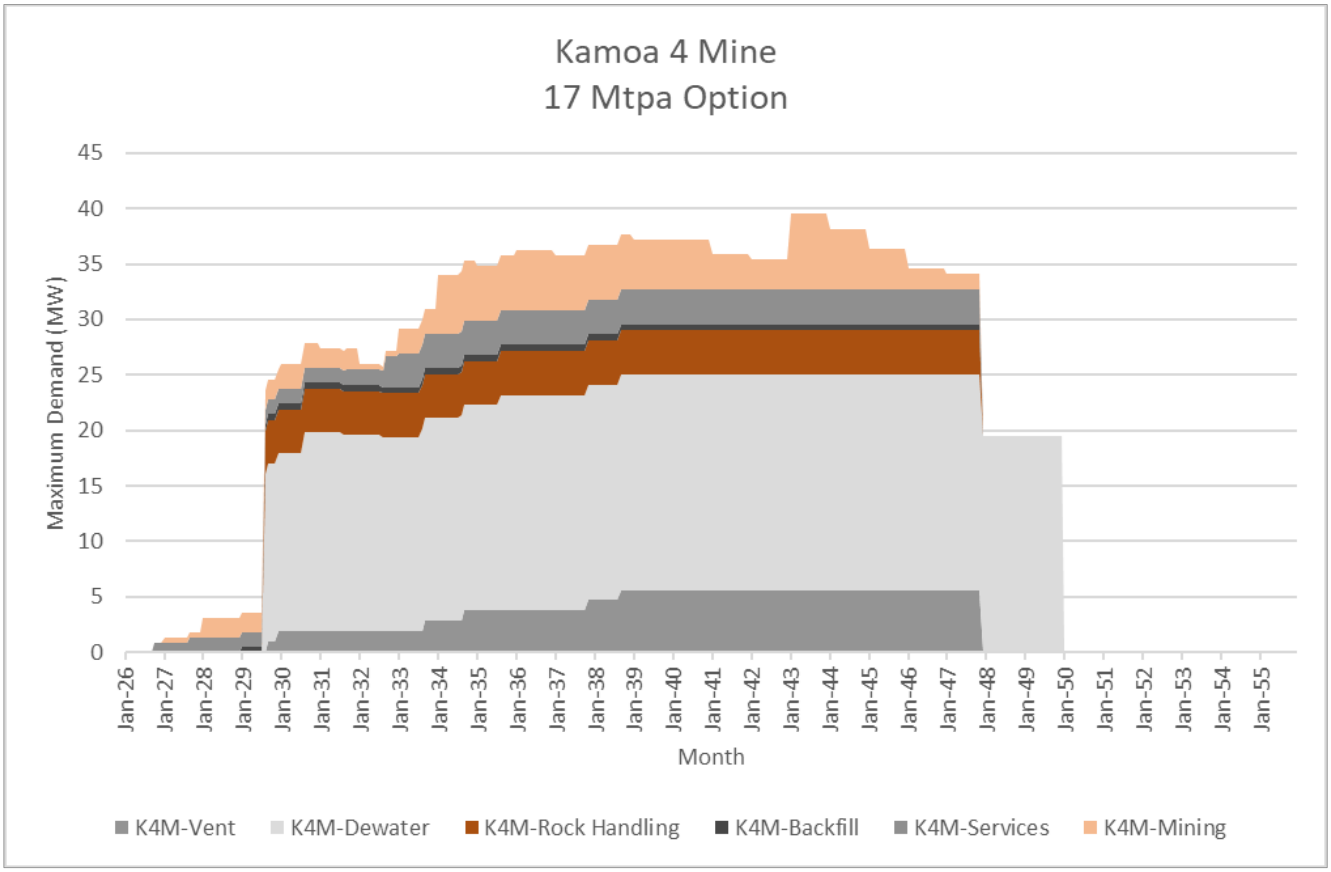
18.4.6.3 33 kV Power Distribution – Kamoa 4

Power will be distributed to Kamoa 4 Mine substations via 33 kV OHLs from the 33 kV Kamoa KCS substation #2. The design allows for an N-1 redundancy to ensure supply reliability.

File Name K4M excel SLD rev A display a substation block diagram for the Kamoa 4 Mine surface and underground substations.

Figure 18.41 indicates Kamoa 4 mine power requirements.

Figure 18.41 Maximum Demand power for the Kamoa 4 Mine



Source: DRA, 2026.

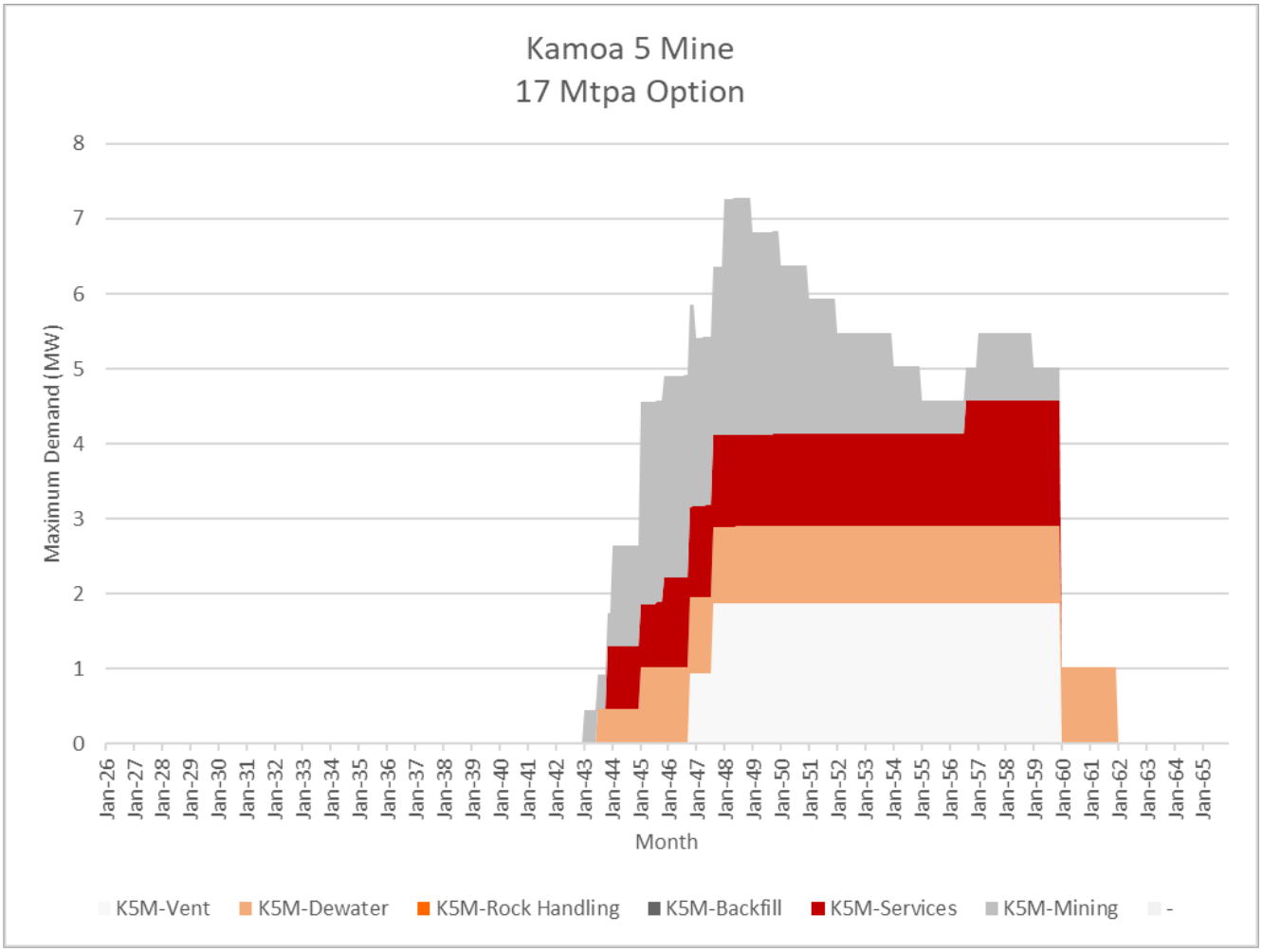
18.4.6.4 33 kV Power Distribution – Kamoa 5

Power will be distributed to Kamoa 5 Mine substations via 33 kV OHLs from the 33 kV Kamoa KCS substation #2. The design allows for an N-1 redundancy to ensure supply reliability.

File Name K5M excel SLD rev A display a substation block diagram for the Kamoa 5 Mine surface and underground substations.

Figure 18.42 indicates Kamoa 5 mine power requirements.

Figure 18.42 Maximum Demand power for the Kamoa 5 Mine



Source: DRA 2026

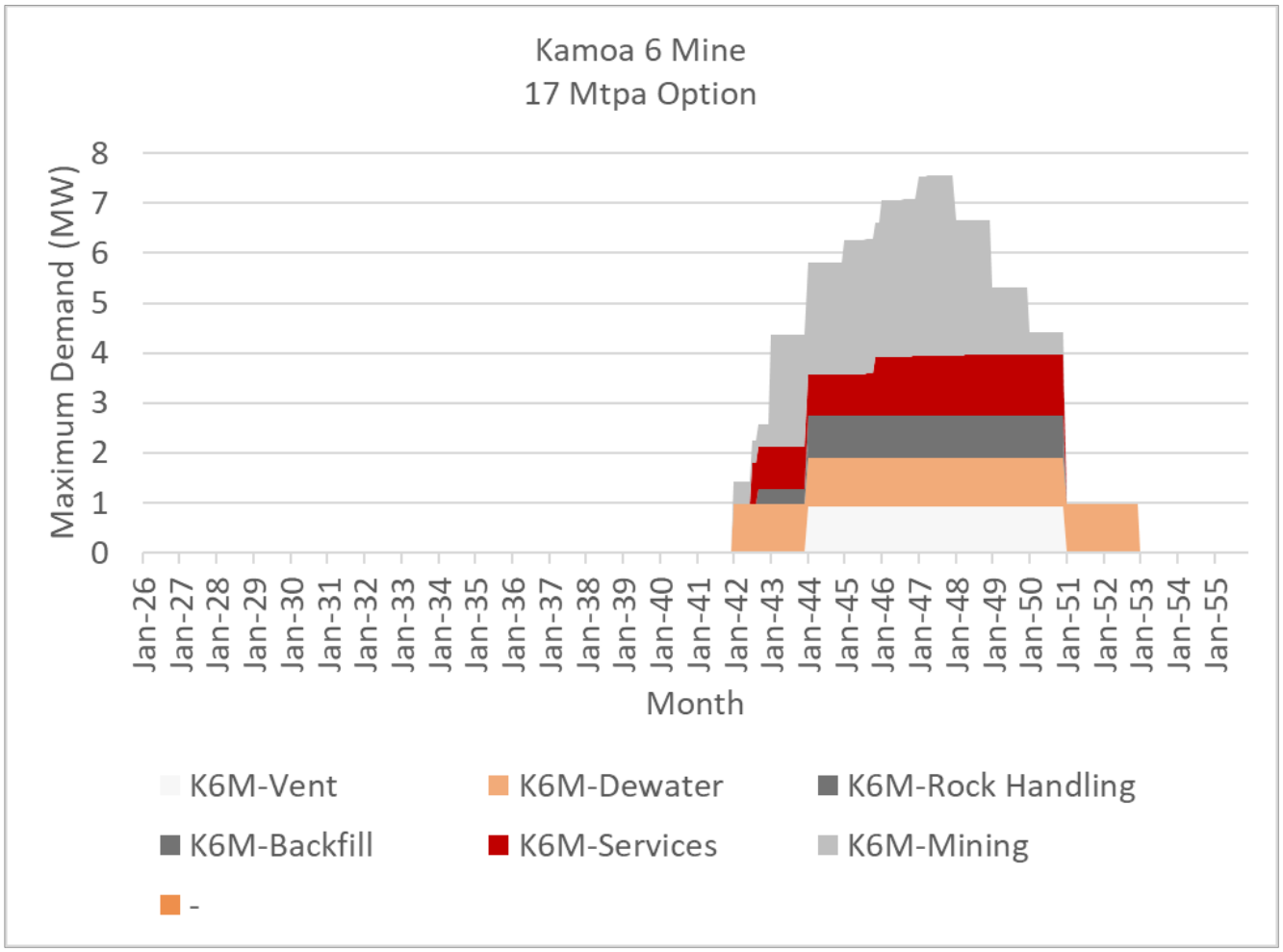
18.4.6.5 33 kV Power Distribution – Kamoa 6

Power will be distributed to Kamoa 6 Mine substations via 33 kV OHLs from the 33 kV Kamoa KCS substation #2. The design allows for an N-1 redundancy to ensure supply reliability.

File Name K6M excel SLD rev A display a substation block diagram for the Kamoa 6 Mine surface and underground substations.

Figure 18.43 indicates the Kamoa 5 mine power requirements.

Figure 18.43 Maximum Demand power for the Kamoa 6 Mine



Source: DRA 2026.

18.4.6.6 Solar PV and BESS

Kamoa is busy with the installation of the following Solar PV and BESS systems by IPPs. The Solar / BESS plants are baseload specified and can therefore also serve as standby supply.

- Kamoa #1 – 30 MW baseload due for commissioning June 2026.
- Kamoa #2 - 30 MW baseload due for commissioning August 2026.
- The following systems are in the planning phase and will be installed at Kakula.
- Kakula #1 – 30 MW baseload planned for commissioning December 2026.
- Kakula #2 – 30 MW baseload planned for commissioning January 2028.
- The following Solar / BESS systems are planned to supply power into the future Kamoa KCS #2 substation.
- Kamoa #3 – 30 MW baseload planned for commissioning December 2029.
- Kamoa #4 – 30 MW baseload planned for commissioning January 2029.

18.4.6.7 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) was commissioned to monitor all MV switchboards, synchronize generators, switch off non-essential breakers and optimize the generator power plant's efficiency. The PMS will be extended for the future additional installations.

Existing generator:

- Kakula M&I substation – 20.8 MW generators.
- Kakula Consumer Sub Gen farm (1) - 20.8 MW generators.
- Konsoko M&I Generator Farm (Kohler) (1) – 7.2 MW.
- Kansoko M&I Generator Farm (Sumec) (2) – 20 MW.
- Kakula Concentrator Generator farm (2) – 72 MW.
- Kamoa Concentrator Generator farm – 48 MW.
- Smelter – 20.8 MW.

18.4.6.8 LV Distribution

The LV Designs are based on existing installation. The MCC distribution voltage will be 690 V for surface and underground infrastructure. Typical LV designs are included in the project Appendixes.

18.4.6.9 Instrumentation and Control Systems

The Instrumentation and control estimate for the study is based on 30% of the electrical estimate.

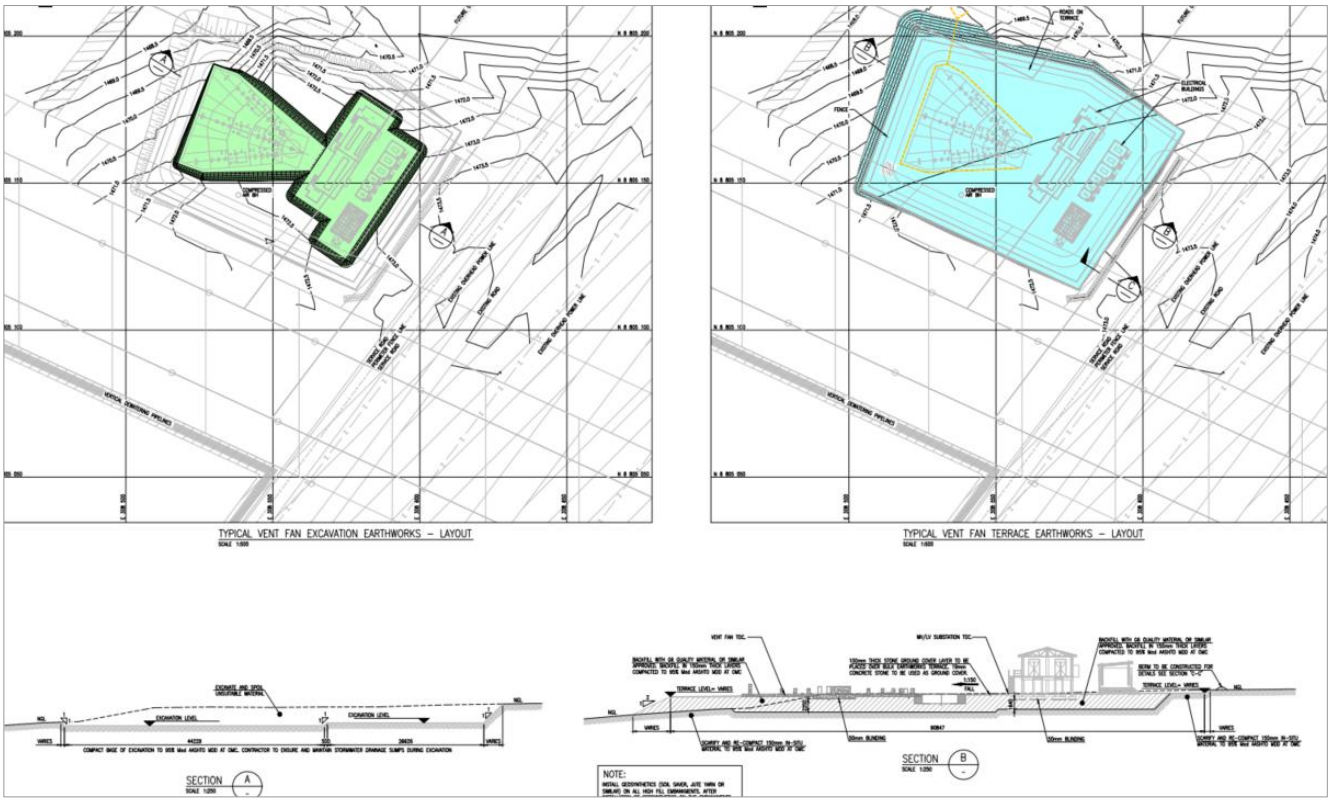
18.5 Ventilation and Cooling infrastructure

The ventilation shafts will consist of upcast and down-cast shafts, refrigeration plants, terraces and supporting infrastructure. The infrastructure includes all earthworks, access roads, civils, raise-bore drilling, electrical and mechanical infrastructure.

18.5.1 Upcast Ventilation Shafts

The upcast ventilation system consists of a tri-fugal exhaust system. Refer to Figure 18.44 for a typical layout of the Upcast ventilation complex.

Figure 18.44 Up-cast Ventilation Shaft - Typical Layout

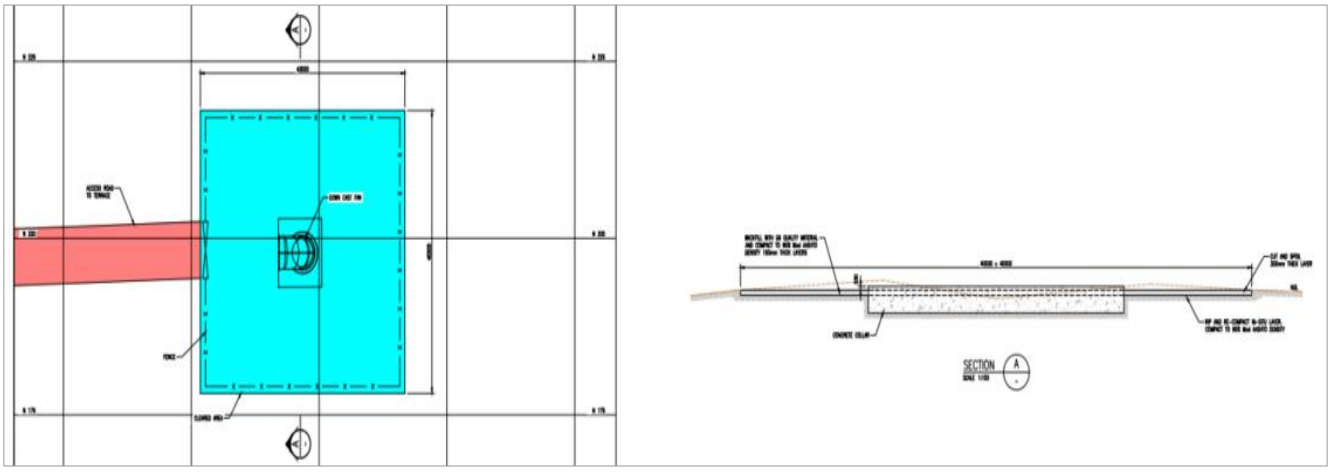


Source: DRA 2026.

18.5.2 Down-cast Ventilation Shaft

The down-cast shaft consists of a terrace and the raise-bore collar civils, terrace capping, and access roads. Refer to Figure 18.45 for a typical layout of the Downcast ventilation complex.

Figure 18.45 Down-cast Ventilation Shaft – Typical Layout

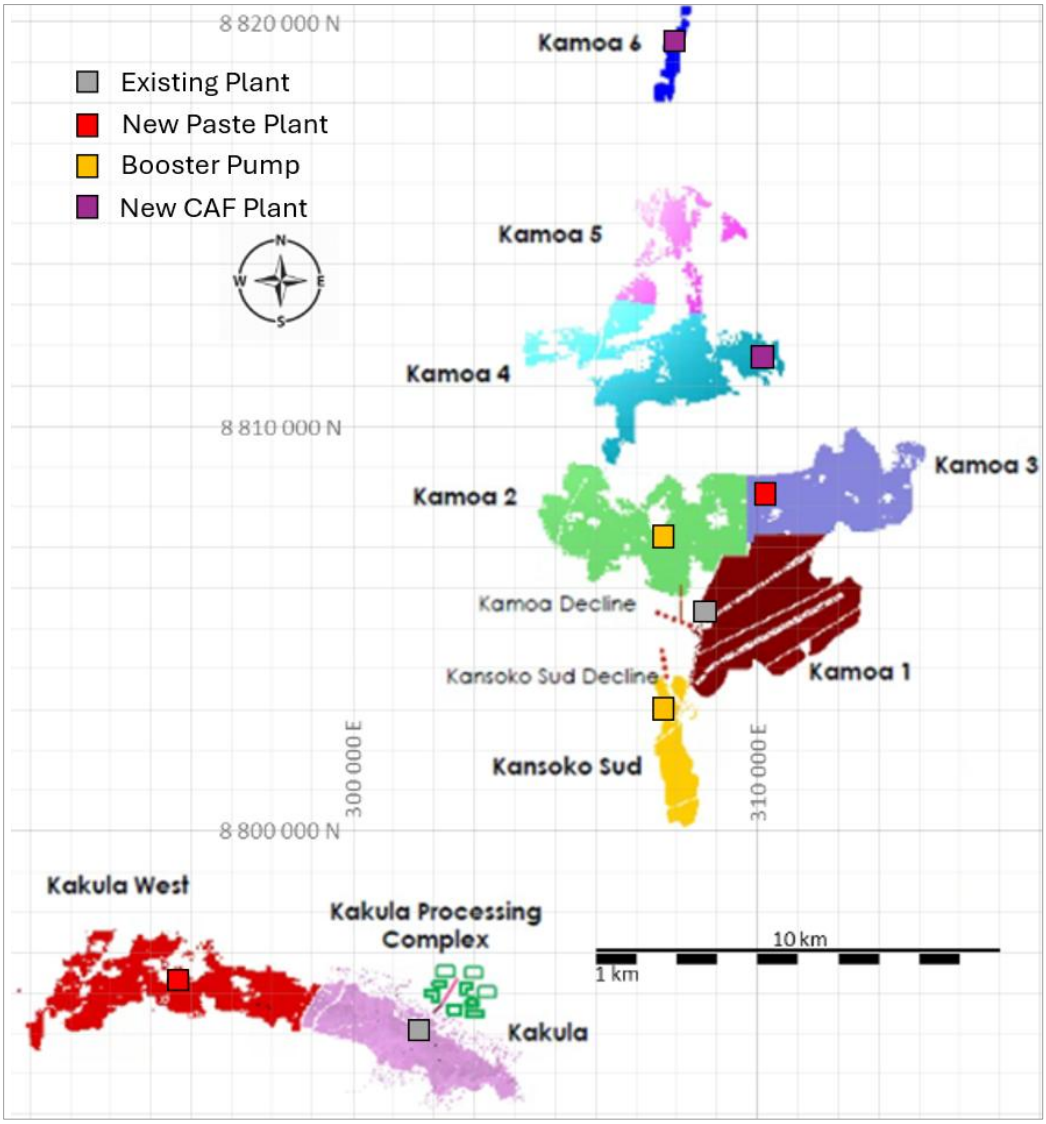


Source: DRA 2026.

18.6 Backfill infrastructure

LOM backfill requirements were estimated by Patterson and Cooke based on mining production profiles. Backfill infrastructure requirements for the project are illustrated in Figure 18.46, and were augmented onto site wide block plan to identify optimum locations for paste plants, booster pump stations and associated overland pipeline routes and borehole positions.

Figure 18.46 LOM Backfill Infrastructure Location Plan



Source: DRA 2026.

18.7 Summary of Services and Facilities, Ventilation Shafts and Backfilling Facilities

A summary of the various facilities, services, ventilation shafts and other related elements pertaining to the entire complex are listed in Table 18.11.

Table 18.11 Summary of Services, Ventilation and Backfilling Facilities

Item No	Description	Dewatering – Surface Settlers (Qty)	Compressor Buildings and Drop-down System (Qty)	Emulsion Storage Facilities and Drop-down System (Qty)	Fuel Oil & Lube Storage Facility and Dropdown System (Qty)	Shotcrete Batching Facility and Dropdown System (Qty)	Up-cast Ventilation Shafts (Qty)	Down-cast Ventilation Shafts (Qty)	Refrigeration Plant (Qty)	Backfill Paste /CAF Plant (Qty)	Backfill Booster Pumpstation (Qty)
1	Kansoko Mine	Existing	N/A	N/A	N/A	N/A	N/A	2	N/A	N/A	N/A
2	Kakula Mine	4	2	3	3	3	5	6	1	Existing	N/A
3	Kamoa 1 Mine	5	2	3	3	3	8	10	4	Existing	N/A
4	Kansoko Sud Mine	1	1	2	2	2	2	2	N/A	N/A	1
5	Kamoa 2 Mine	4	3	3	3	3	6	4	N/A	N/A	1
6	Kamoa 3 Mine	4	3	4	4	4	6	12	N/A	1	N/A
7	Kamoa 4 West Mine	1	3	3	3	3	7	12	N/A	1	N/A
8	Kakula West-West Mine	8	2	2	2	2	11	12	1	1	N/A
9	Kamoa 5 Mine	1	2	1	1	1	3	4	N/A	N/A	N/A
10	Kamoa 6 Mine	1	1	1	1	1	1	2	N/A	1	N/A

18.8 Kamoa-Kakula Underground Infrastructure Overview

18.8.1 Dewatering

Mining return water, from underground mining operations and groundwater inflow into mined-out areas, is managed through underground sumps, dams and pumpstations. Submersible pumps located in the working areas will transfer dirty water to a series of transfer dams and pumpstation which discharge dirty water into vertical transfer and multistage dams for subsequent discharge to surface settling dams discussed in Section 16.

18.8.1.1 Submersible and Sump Pumps

Submersible sump pumps located at the working faces, will pump dirty water to transfer dams. These are temporary installations, adapting to the advancing mining faces.

18.8.1.2 Transfer Dams and Pumpstations

Transfer dams and pumpstations are strategically located across the mine footprint to receive dirty water from the stopes for discharge to vertical transfer and multistage dams. They consist of up to six pump trains (which includes a standby pump train), each train delivering up to 200 l/s. Pumpstations with lower delivery static heads will have a single pump or two pumps per train, i.e., a single pump has a total dynamic delivery head (TDH) of 75m and is powered by a 315 kW electric motor. Table 18.12 summarizes specifications for the three types of transfer pump stations for this project.

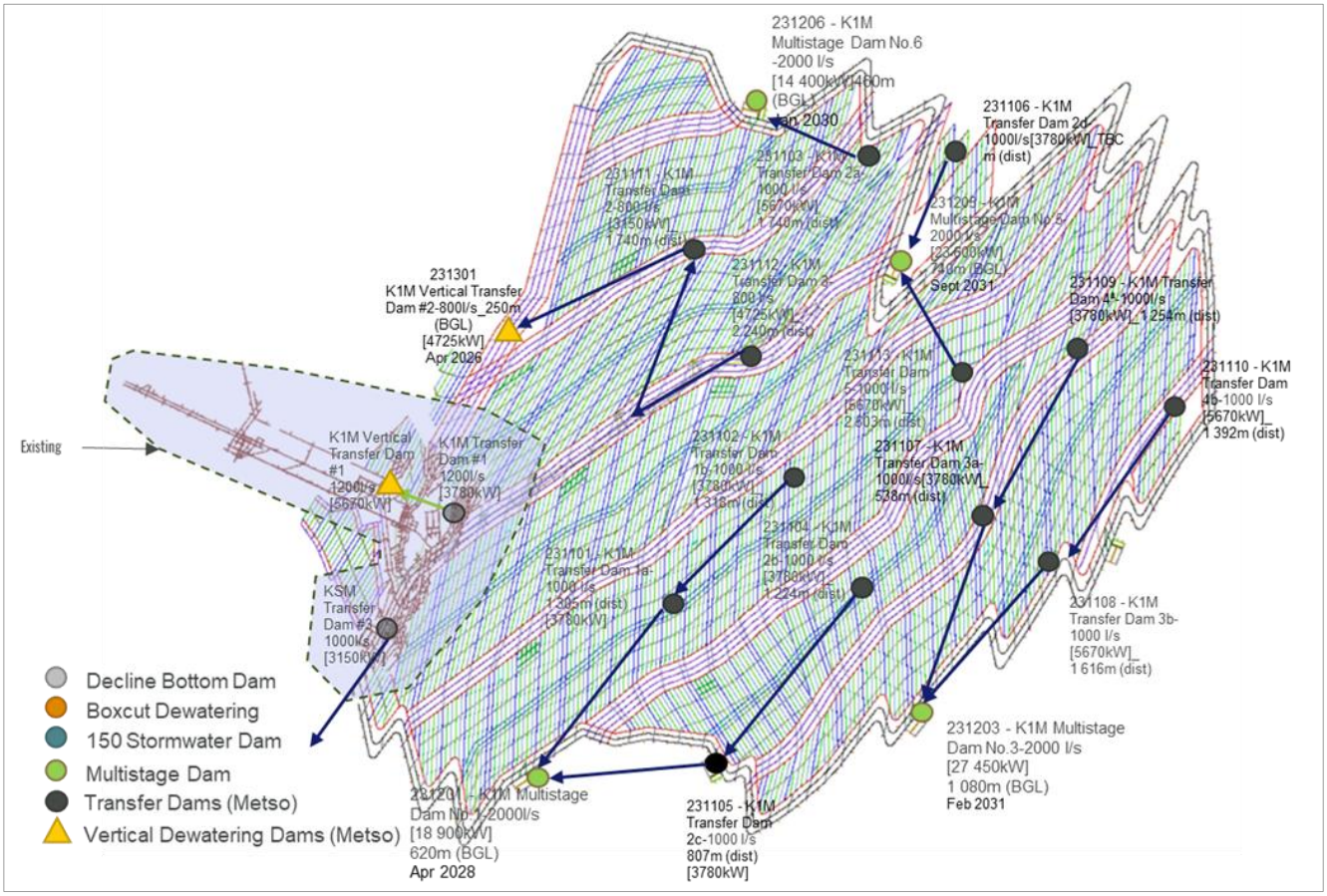
Table 18.12 Transfer Pumpstation Specifications

Type	Pumps per Train	Max. TDH (m)	Installed Power (kW)*
1	1	75	1,890
2	2	150	3,780
3	3	222	5,670

Note: *Installed kW are based on a six-pump train pump station.

Throughput for transfer pumpstation(s) is matched to the pumping capacity for the downstream vertical transfer or multistage pumpstation. Figure 18.47 illustrate cases for transfer dams in parallel discharging into a common multistage dam. The figure also shows two transfer pumpstations in tandem, for cases where the total dynamic delivery head to the multistage dam cannot be achieved by one pumpstation due to THD constrains.

Figure 18.47 Kamoa 1 UG Mine Pumpstation Location Plan

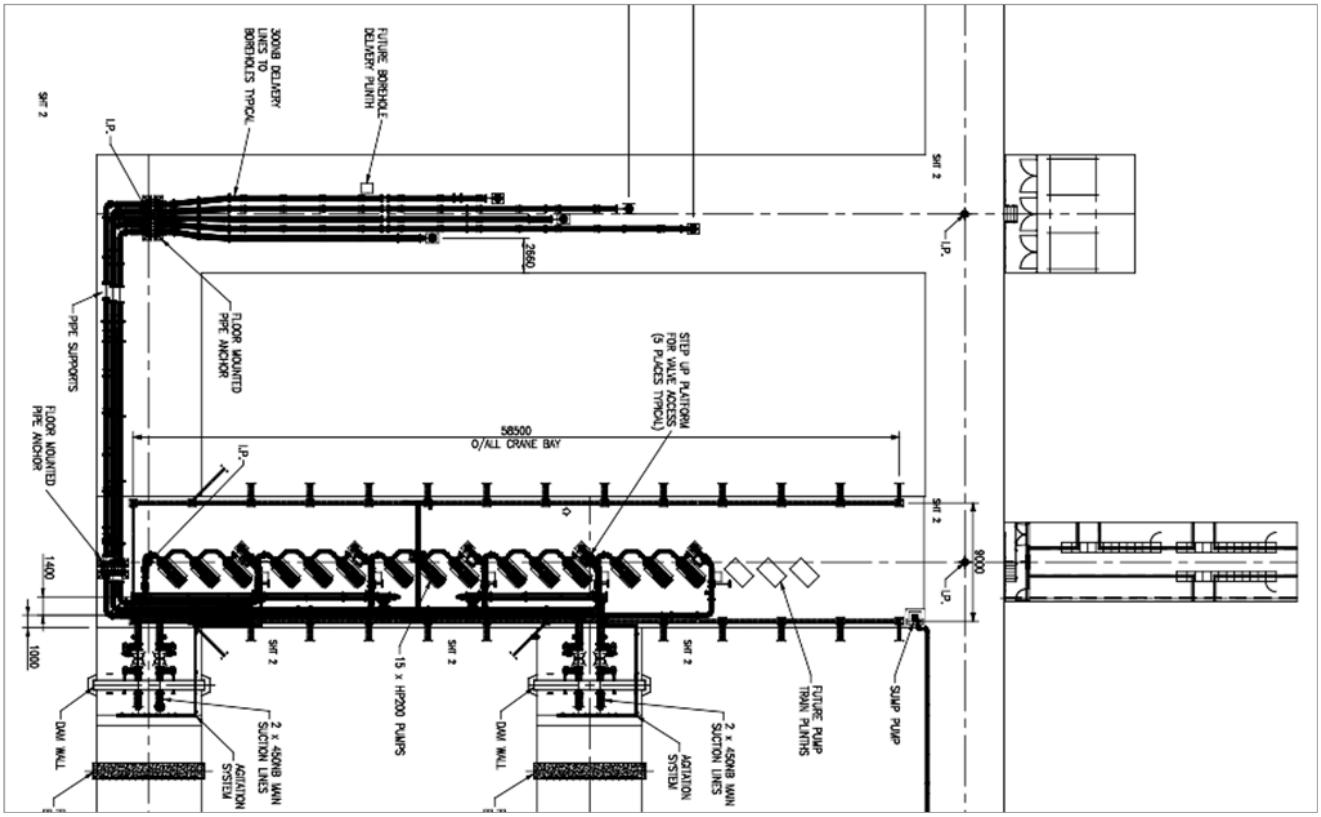


Source: DRA 2026.

18.8.1.3 Vertical Transfer Dams and Pumpstations

Vertical transfer dams receive dirty water from transfer dams, in Section 18.8.1.2, for subsequent pumping and discharge to surface settling dams. They consist of the similar centrifugal pumps used in transfer with up to six pump trains (which includes a standby pump train), each train delivering up to 200 l/s. The maximum total dynamic delivery head for vertical transfer pumpstations is 222m achieved via three pumps in series per pump train. Most vertical pump stations consist of Type 3 installation, in Table 18.12 and Figure 18.48 due to the average static head exceeding 150m. The difference between transfer and vertical transfer pumpstations is that later pump from underground to surface through individual, 300NB (nominal bore) delivery pipelines installed in vertical boreholes.

Figure 18.48 Typical Vertical Transfer Pumpstation Layout



Source: DRA 2026.

18.8.1.4 Multistage Dams and Pumpstations

Pump stations located deeper than 220m below surface will be equipped with high-lift multistage centrifugal pumps, suitable for pumping clear water for discharge to surface settling dams. Gravity separators and de-gritter units will be installed upstream of clear water storage dams to settle and remove grit from dirty water before pumping clear water to surface.

Multistage pumpstation capacities for the project range from 1000 l/s to 2250 l/s made up of multiples of 250 l/s multistage pumps discharging into rising mains installed in boreholes, illustrated in Figure 18.49. The current Kakula Multistage Dam No.1 West comprise two (2), 600NB rising mains, each connected to three (3) multistage pumps in parallel configuration. All pumpstations will be equipped with a single standby pump. Table 18.13 summarizes specifications for the three types of multistage pump stations for this project.

Table 18.13 Multistage Pumpstation Specifications

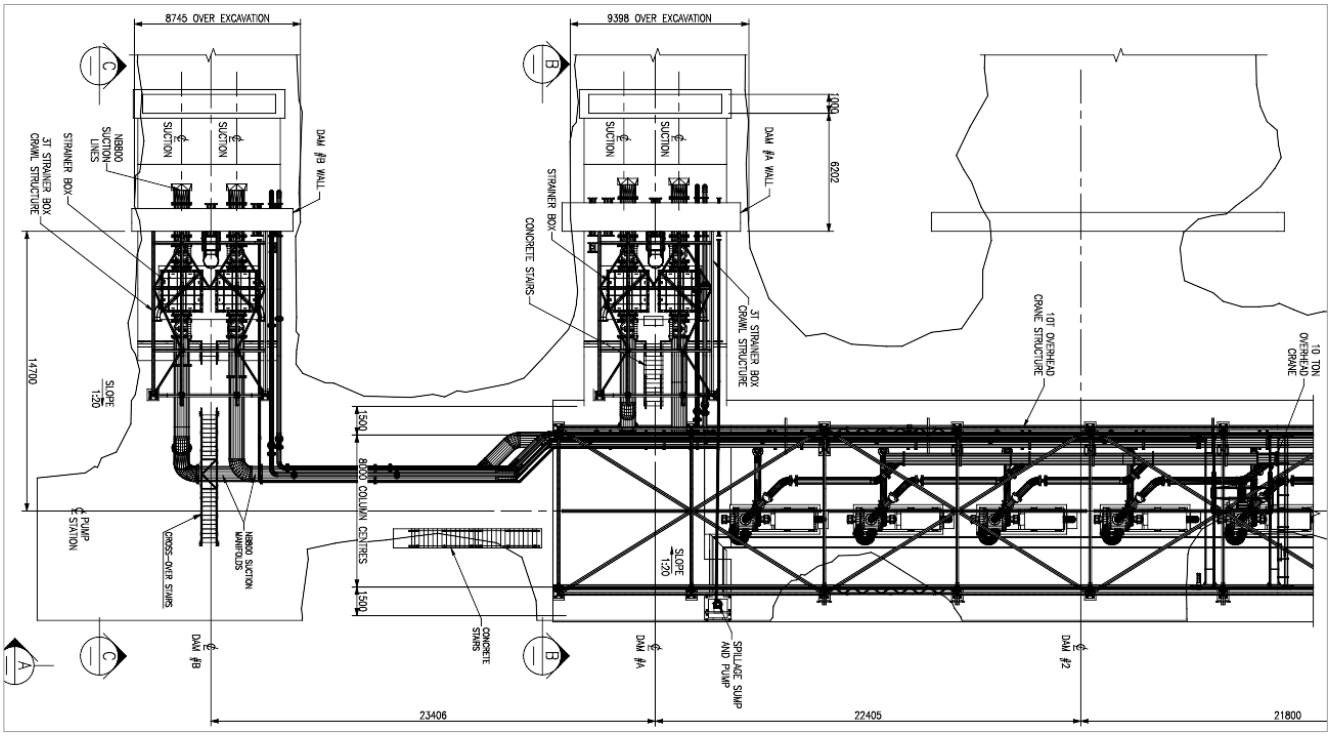
Type	Pumpstation Capacity (l/s)	No. of Installed Pumps	No. of 600NB Rising Mains
Type 1	1000	4	1
Type 2	1500	6	2
Type 3	2250	9	3

Standard electric motors sizes in Table 18.14 were selected for the various pump station depth below surface.

Table 18.14 Multistage Dam Electric Motor kW Ratings

Depth Below Surface (m)	Motor Rating (kW)
225 - 500	1,600
500 - 700	2,100
700 - 850	2,600
850 – 1,050	3,050
1,050 – 1,500	5,200

Figure 18.49 Typical Three Pump Parallel Installation



Source: DRA 2026.

18.8.1.5 Basis of Dam and Pumpstation Location

Positioning of underground mine dewatering infrastructure involved integrating estimated Yearly groundwater inflow volumes and with mine production/advancement planning. The Deswik models for each mine was used to identify optimum locations for vertical transfer and multistage transfer pumpstations. The following key requirements were used for the placement of vertical transfer and multistage dams on the mine plan:

- Geological structures – all UG to surface transfer dams and pumpstations are the lifeline for dewatering the mine and are installed (as much as possible) on the underground mine boundary top avoid areas with major faults, shear zones, or fractures rock that could cause leakage or structural failure usually consistent with mined-out areas.
- Pumping Capacity – Provision of the required pumping capacity matching or exceeding the modelled ground water inflows in coordination with the advancing mining face to handle water inflow for newly opened up areas i.e., mainly due development and stoping.

Transfer, vertical transfer and multistage dams and pumpstations are permanent installations and seldom shift with advancing mining faces.

18.8.1.6 Pumpstation auxiliary equipment

Pumpstations are equipped with auxiliary systems including but not limited to the following:

- Suction and discharge pipework complete with strainer boxes, manual and automated isolation valves and surge prevention facilities.
- Valve maintenance platforms, overhead travelling cranes, crawl beam structures for maintenance and repair requirements.
- Structural support steelwork for overhead cranes and pipes.
- Spillage sump pumps.
- Instrumentation for operational control.
- Electrical facilities including medium voltage substations, transformers and low voltage substations.

18.8.2 Rock handling

The main transportation of waste rock and ore from underground workings to surface is through hauling / dump trucks only or a combination of dump trucks and decline shaft conveyor belts. Rock is transferred from working faces to multiple bunkers i.e. open, mined-out areas for stockpiling, rehandling using LHDs, and transportation to surface stockpiles using dump trucks or conveyor belts. Table 18.15 summarizes the ore handling philosophy for each mine, current status and life-of-mine requirements.

Table 18.15 UG Surface Rock Handling Criterion per Mine

Item No	Boxcut / Decline Shaft	Rock Handling System	Status and LoM Requirements
1	Kansoko Mine	Hauling	Existing. Hauling to be extended per new mine plan
2	Kakula Mine		
2.1	Kakula Mine North Decline	Decline Conveyor and Hauling	Existing. Hauling to be extended per new mine plan
2.2	Kakula Mine South Decline	Hauling	Existing. Hauling to be extended per new mine plan
3	Kamoa 1 Mine	Decline Conveyor and Hauling	Existing. Six new conveyors and trucks tips required Hauling to be extended per new mine plan
4	SUD Mine	Hauling	Existing. Hauling to be extended per new mine plan and via new SUD boxcut (instead of Kansoko Boxcut)
5	Kamoa 2 Mine:		
5.1	Kamoa 2 East Mine	Hauling	New
5.2	Kamoa 2 West Mine	Hauling	New
6	Kamoa 3 Mine	Decline Conveyor and Hauling	New. One main decline shaft conveyor and two silos required
7	Kamoa 4 Mine		
7.1	Kamoa 4 North Mine	Hauling	New
7.2	Kamoa 4 South Mine	Hauling	New
7.3	Kamoa 4 West Mine	Hauling	New
8	Kakula West Mine		

Item No	Boxcut / Decline Shaft	Rock Handling System	Status and LoM Requirements
8.1	Kakula West-West Mine	Decline Conveyor and Hauling	New. Two main decline shaft conveyors and two silos required
8.2	Kakula West-East Mine	Hauling	New
9	Kamoa 5 Mine	Hauling	New
10	Kamoa 6 Mine	Hauling	New

Conveyor systems consist of 1,500 mm wide belts rated for 2,000 tph, ore passes and silos equipped with truck tipping facilities, bulkhead discharge arrangements, apron feeders and sacrificial conveyors for transfer of rock from underground dump trucks to main decline belts discharging onto surface stockpiles.

18.8.3 Services and utilities

18.8.3.1 Compressed air

Compressed air for underground refuge chambers is supplied from compressor houses located on surface through borehole piping. The design for each compressor house includes three air cooled, oil-free compressors, two operating and one on standby, an air receiver, overland and borehole piping. Ancillary equipment including manual and automatic control valves, pressure relief valves, silencers, pressure relief valves will be installed in the compressed air piping system to regulate air supply pressure and volume to the refuge chambers.

18.8.3.2 Firewater

Firewater will be supplied to fire sprinkler systems for underground conveyor belts (where applicable). Auxiliary firefighting equipment for underground conveyors includes:

- Firewater pressure reducing stations.
- Valve stations.
- Fire water reticulation piping including isolation valves.
- DCP and CO_2 fire extinguishers.
- Linear heat detection systems for activating sprinkler and deluge systems.
- Signage.
- Fire panels.
- Smoke detectors, alarms, xenon beacons, lamp units etc.

Fire detection and suppression systems complete with ancillary equipment will be developed, in accordance with the Kamoa-Kakula fire protection standard and designed and supplied by specialist turnkey contractors.

18.8.3.3 Potable Water

Potable water is currently supplied from treated borehole water pumped to a storage tank at Kamoa 1 boxcut for downstream reticulation to various end-user points. A new borehole well-field will be drilled at Kamoa 4's West Scarp fault zone to provide potable water for new mines.

18.8.3.4 Service Water

Service water for underground mining and engineering operations is supplied from recirculated, filtered mine return water. All new mines for this project are water positive and will adopt similar recirculation strategies to limit freshwater intake.

18.8.3.5 Emulsion

Emulsion vertical drop systems and associated underground storage and handling.

18.8.3.6 Fuel and Lubricants

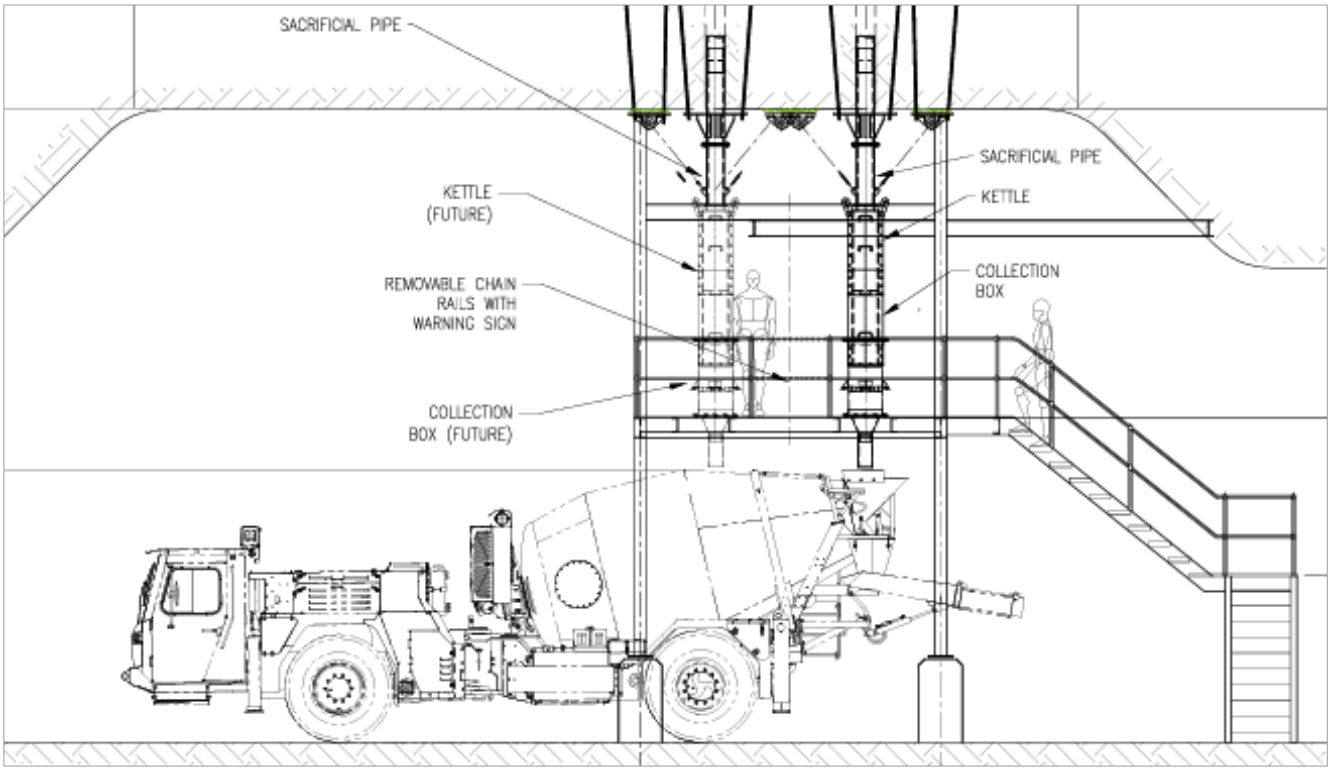
Vertical fuel and lubricants drop systems terminate at bulk storage tanks located at underground main workshops Diesel and lubricant dispensers are installed in cubbies along the main refueling haulage so mobile machinery can be refueled outside of the main underground workshop. Drawing L-20000-00585-01 shows the arrangement of bulk storage tanks and refueling facilities.

Spillage from the refueling and lubrication bays is channeled to a collection sump equipped with an oil separator. Clean water from the oil water separator is pumped into the mine wide return water handling system.

18.8.3.7 Shotcrete

Vertical drop shotcrete slicklines discharge into a dedicated kettles and hoppers, which hold the shotcrete before discharging into a shotcrete carrier for downstream application, as illustrated in Figure 18.50.

Figure 18.50 Underground Shotcrete Facilities



Source: DRA 2026

Auxiliary facilities at the underground shotcrete handling plant include:

- A silt trap, sump and pump for handling dirty water from hosing down and slickline flashing.
- Fibre and other additives storage cubby complete with a crawl beam, lifting tackle and hoist for loading fibres and additives into collection boxes/hoppers.
- Shotcrete discharge funnel for transferring shotcrete into the mobile shotcrete utility vehicle.

Underground shotcrete handling facilities and located adjacent to the underground main workshops in the same cluster for emulsion, fuel and lubrication drop down systems.

18.8.4 Underground Workshops

18.8.4.1 Trackless Mining Machinery Main Workshops

A main underground workshop will be established at each mine, when the tramming distance from underground workings to the main workshop on surface is deemed excessive, usually exceeding 1500m (one-way tramming distance). Additional underground main workshops will be established to maintain the tramming radius between workings and workshops to 1500m.

Where possible, redundant workshops will be relocated to new mining areas depending on the mining production planning and condition of workshop infrastructure.

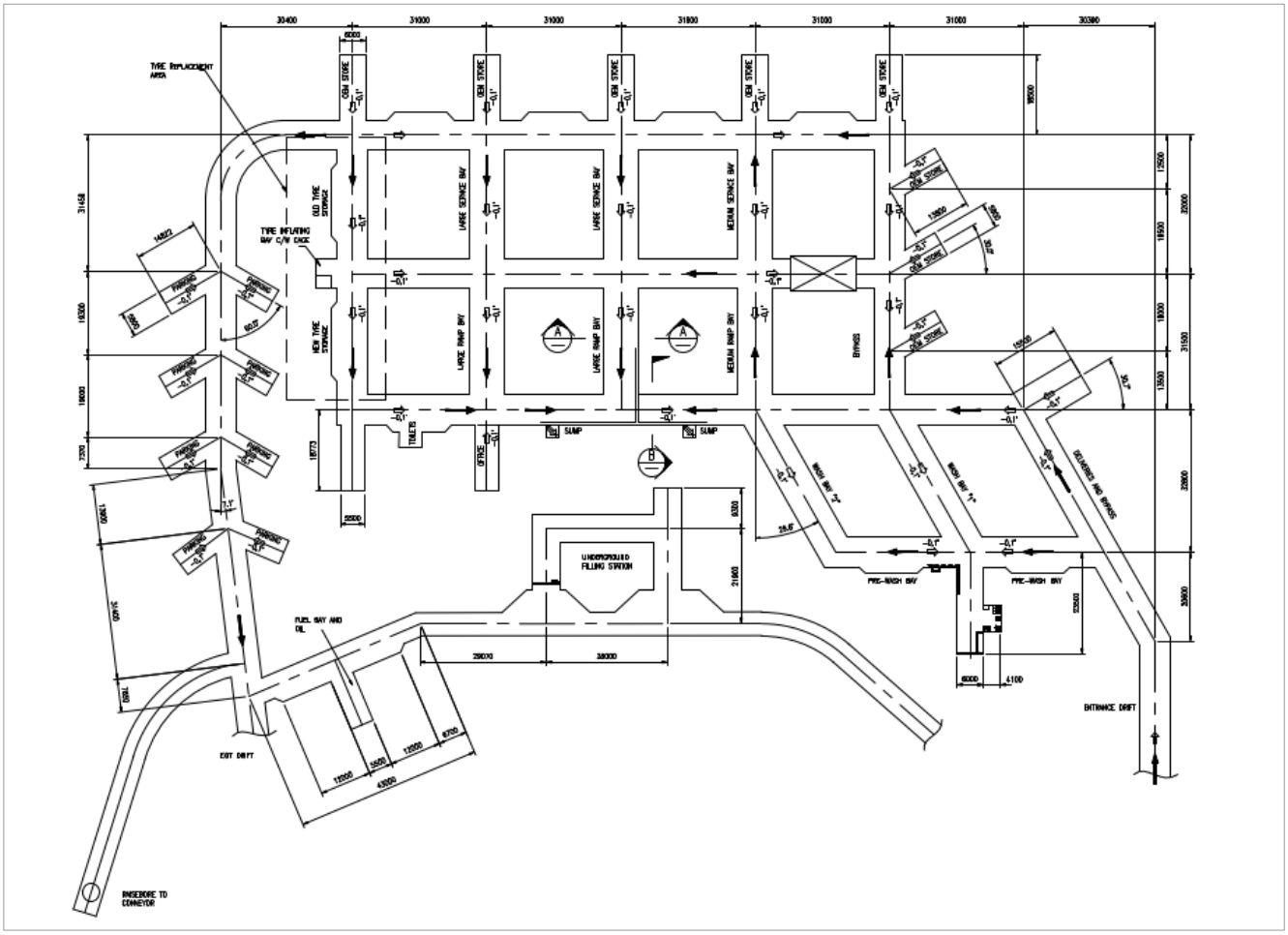
Maintenance services for 200, 250, 500, 1000, 1200, 3600-hour intervals (whichever is applicable to drill rigs, LHDs and dump trucks) will be carried out at the main underground workshop. The following are provided at main underground workshops to facilitate mobile machinery maintenance requirements:

- Wash bays for high-pressure cleaning machinery before servicing.
- Silt trap, sump and oil/water separator unit and associated oil handling equipment.
- Inspection bays equipped with service steel ramps and crawl beams.
- Flat service bays with a 10-ton overhead cranes for major repairs.
- Flat repair bay with a 25-ton overhead crane for equipment rebuilds and major components replacements.
- Service bay equipped with 5-ton overhead cranes.
- Tyre storage bay complete with tyre handling facilities.
- Hydraulic and pneumatic shop.
- Lubrication and refueling bay enabling mobile machinery to by-pass the main workshop area.
- Parking bays for various machinery.
- Lockable stores for low value consumables for various mobile machinery.
- Offices, amenities and work bench areas.

Ancillary facilities including fire detection and suppression systems will be provided. A part of the workshop will be barricaded into a stores area with a 5-ton overhead crane for heavy components. Drawings L-200000-00582-01 illustrates typical main underground workshop layout and details of overhead travelling cranes. Underground main workshops will be located along the main access decline shafts, in geologically stable areas with ease of access and central to active working areas. All workshops will be connected to ventilation return airways for fume management. Refer Figure 18.51.

Specific workshops at specific areas will differ pending productions requirements and equipment fleet.

Figure 18.51 Typical underground workshop



Source: DRA 2026

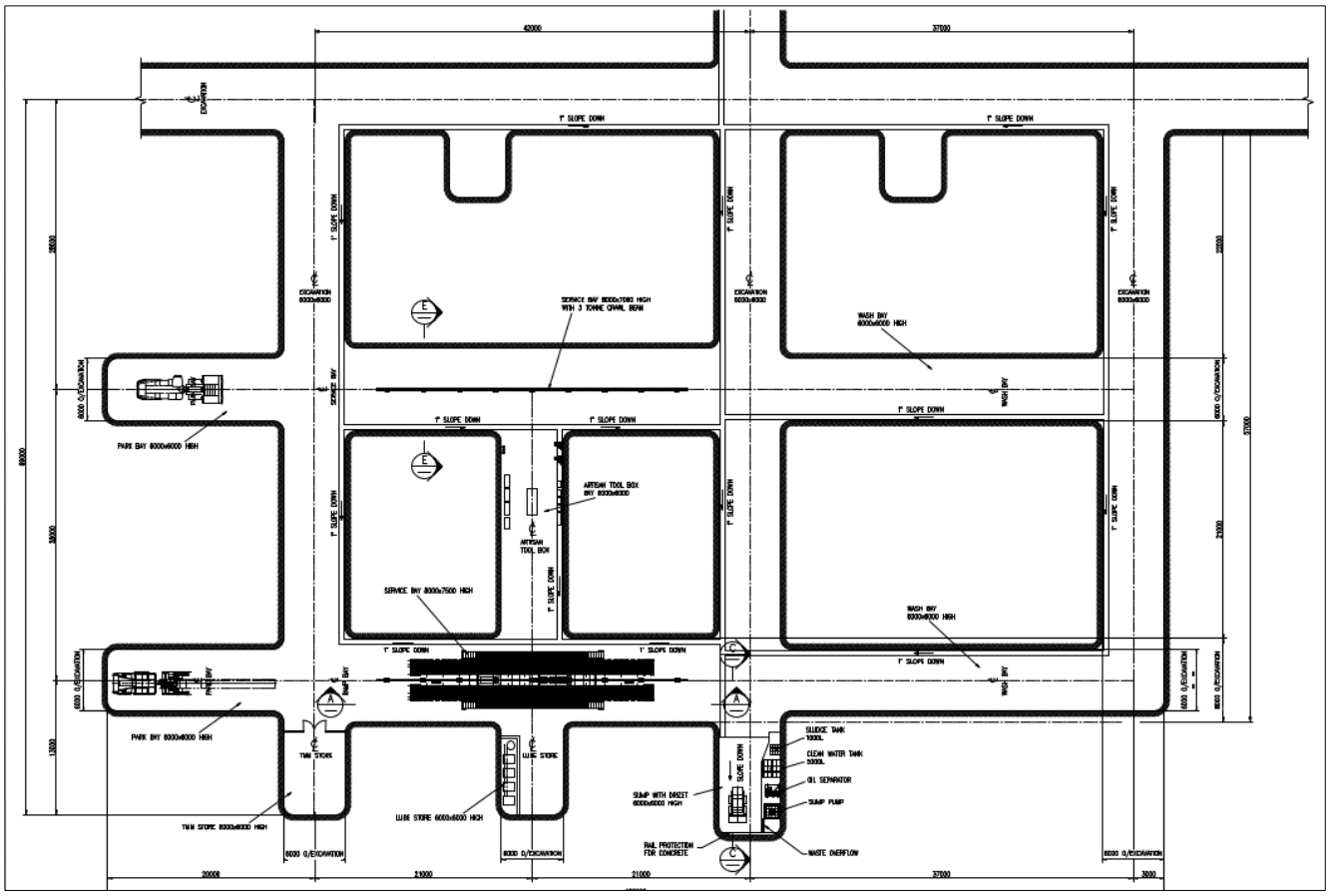
18.8.4.2 Trackless Mining Machinery Satellite Workshops

Satellite workshops will be established in mined out areas, at least 100 m but not more than 500 m from active working faces. Daily and weekly service checks and minor repairs will be performed at these workshops, which also provide end-of-shift parking bays for mobile mining machinery local to the working faces. Satellite workshops will be relocated as the mine advances and will be equipped with the following facilities:

- One inspection bay incorporating a steel ramp and a 3-ton crawl beam.
- Two service bays, one with a 3-ton crawl beam and another a plain bay.
- A lockable store for storing low value consumables.
- Mobile machinery parking bays.
- Lubrication storage bay.
- Wash bay complete with oil water separator unit and associated oil handling facilities.
- Office, work bench areas and other amenities for workforce.

Provision will be made at the workshop for a refueling bay which will be via a Utility Vehicle equipped with the necessary cassette. Refer Figure 18.52.

Figure 18.52 Typical underground satellite workshop



Source: DRA 2026

18.9 Mine Specific Infrastructure

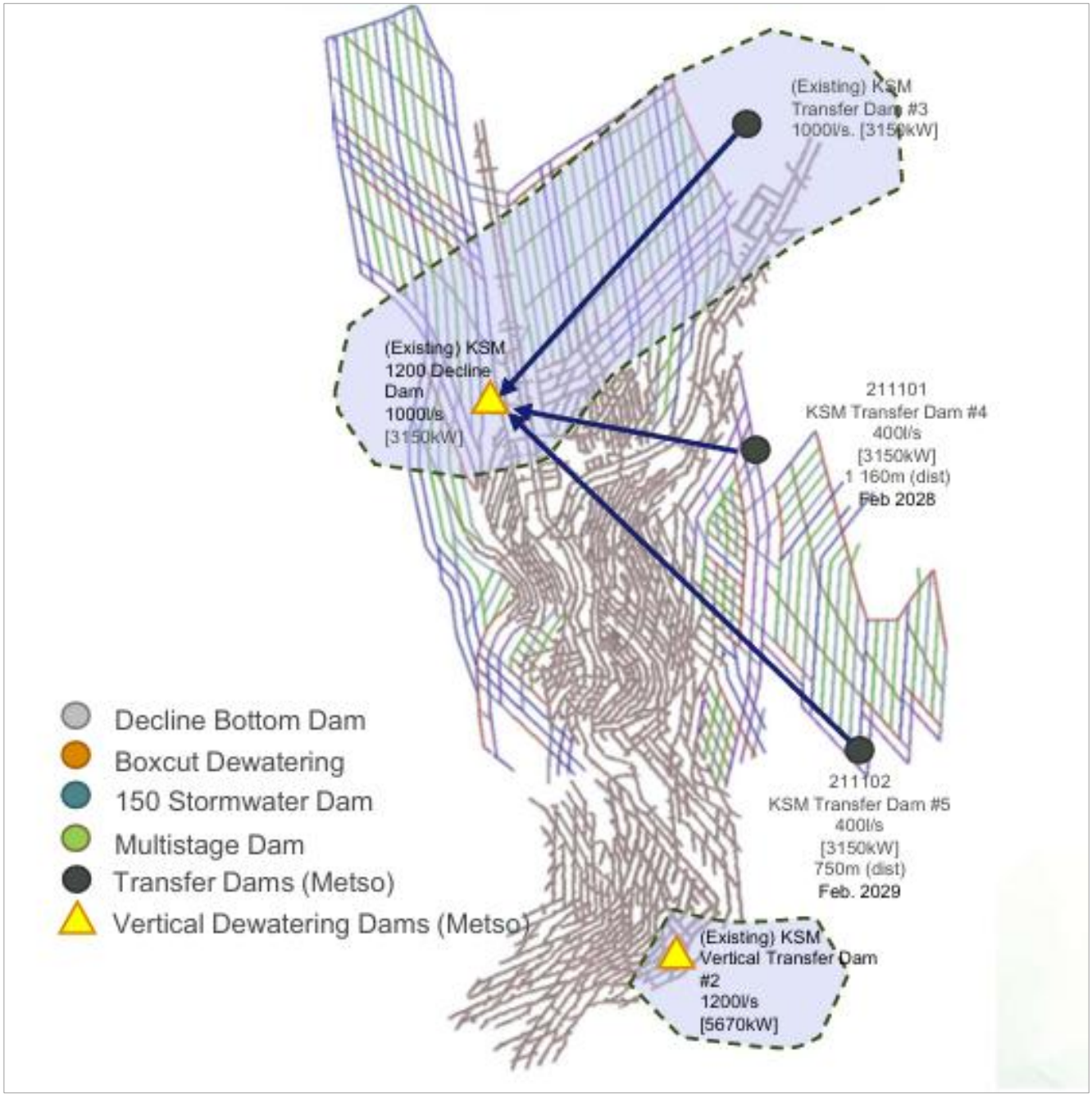
This section covers key infrastructure for each mine and should be read in conjunction with the overall block plan for the whole mining complex, and individual surface and underground block plans and layouts for each mine.

18.9.1 Kansoko

18.9.1.1 Kansoko Mine Dewatering Infrastructure

Kansoko mine currently has two vertical transfer dams and pumpstations discharging to surface. The KSM 1200 Decline Dam has an installed capacity of 1,000 l/s, (i.e., 800 l/s duty and 200 l/s standby capacity) and is located on the upper portion of the orebody. Two new, 400 l/s (each) transfer pump stations, No. 4 and No.5, are required for Kansoko's LOM plan as illustrated in Figure 18.53.

Figure 18.53 Kansoko UG Mine Pumpstation Location Plan



Source: DRA 2026

The estimated LOM groundwater inflow is 126 l/s in 2028. The installed pumping capacity from underground to surface exceeds the estimated peak water, therefore no new vertical transfer pumpstation is required for Kansoko mine as summarized in Table 18.16.

Table 18.16 Kansoko UG Mine Dewatering Capacity Summary

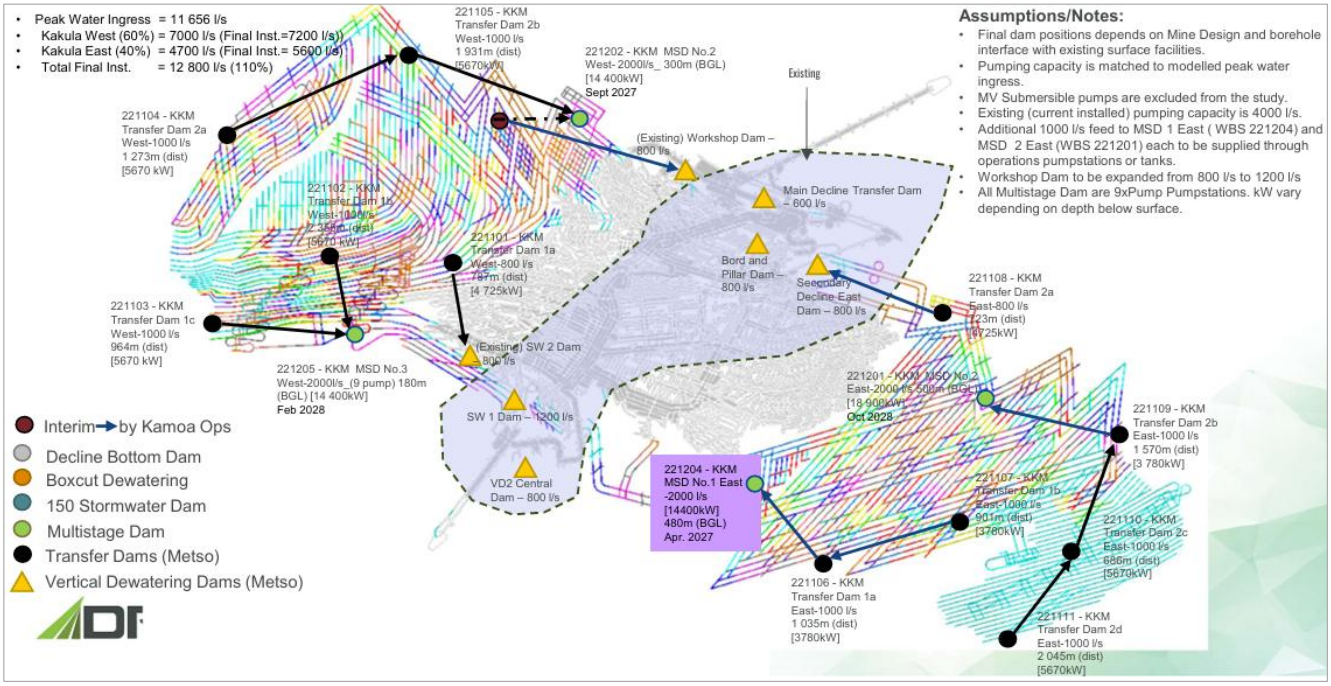
Peak LOM Inflow (l/s)	Installed Capacity (l/s)	Additional Capacity Rqd. (l/s)	Planned Additional Capacity (for Installation) (l/s)	Margin (Safety Margin)
126	800	0,000	0,000	6.35

18.9.2 Kakula Mine

18.9.2.1 Kakula Mine Dewatering Infrastructure

Kakula mine's installed pumping capacity from underground to surface is approximately 4,400 l/s and the estimated LOM peak ground water inflow is 11,600 l/s. The required additional pumping capacity of 8,000 l/s will be achieved through four additional multistage and associated transfer dams and pumpstations indicated in Figure 18.54.

Figure 18.54 Kakula UG Mine Pumpstation Location Plan



Source: DRA 2026

Table 18.17 summarizes peak inflows and required additional pumping capacity over the Kakula LOM plan. The final installed capacity of 12,400 l/s will exceed the estimated peak inflow with a safety margin of approximately 10%.

Table 18.17 Kakula UG Mine Dewatering Capacity Summary

Peak LOM Inflow (l/s)	Installed Capacity (l/s)	Additional Capacity Rqd. (l/s)	Planned Additional Capacity (for Installation) (l/s)	Margin (Safety Margin)
11,600	4,400	7,200	8,000	1.1

18.9.3 Kamoa 1 Mine

18.9.3.1 Acid Mine Dump and Acid Water Treatment Facilities.

The acid generating mine waste material will be hauled and stockpile at on centralized lined stockpile area, located in proximity of the existing Kamoa1 surface infrastructure area and the Kamoa phase 3 concentrator.

18.9.3.2 Acid Mine Waste Dump Stockpile

The AMD Stockpile will consist of a 110Ha, HDPE lined stockpile. The intent is to divert all the clean stormwater (non-contact) past the proposed stockpile area. All the run-off water from the stockpile will be collected and diverted to two lined acid water pollution control dams, from where the water will be treated to a standard, where the water can be released into the environment.

The run-off water from the stockpile will be collected and diverted to two lined acid water pollution control dams, from where the water will be treated to a standard, where the water can be released into the environment.

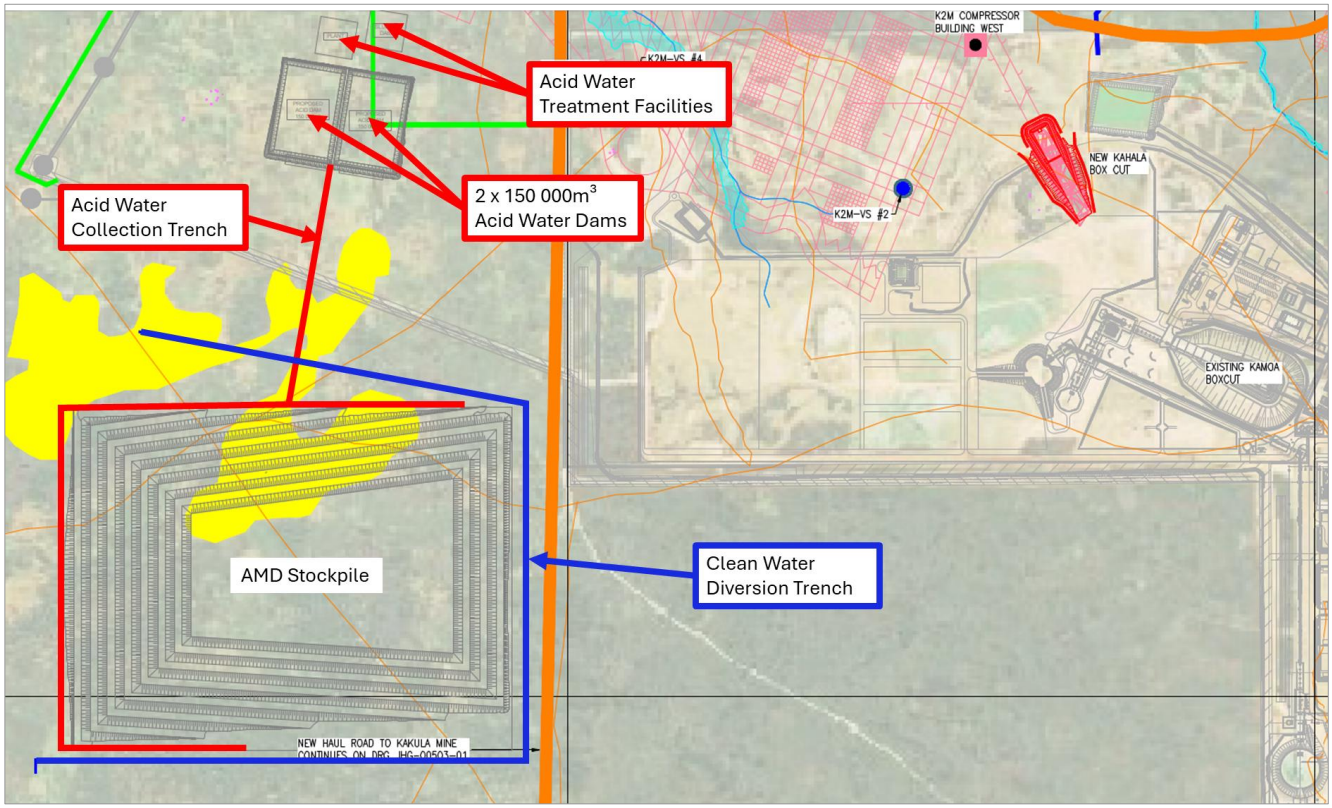
The run-off water from the stockpile will be collected and diverted to two lined acid water pollution control dams, from where the water will be treated to a standard, where the water can be released into the environment.

18.9.3.3 Acid Mine Waste Dump

The Acid-Generating Mine-Waste Dump (AMD) design is not complete. The Geotechnical design is required before the design of the dump can commence.

The AMD stockpile design is based on the existing KPS Stockpile and consists of a HDPE lined stockpile shaped terrace. The total area required for the AMD Stockpile is 110 Ha. Due to the size, the Lining will be phased as the stockpile grows. A plan showing the locations of the AMD Stockpiles is shown in Figure 18.55.

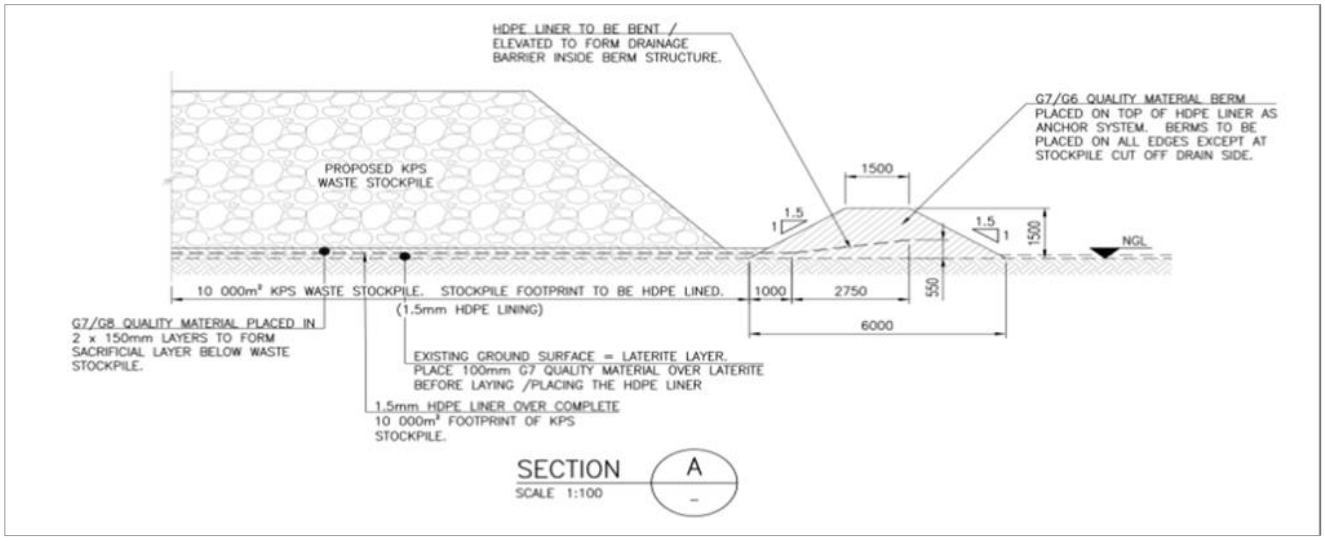
Figure 18.55 AMD Stockpile, Lined Drains, Acid Water Dams and Acid Water Treatment Facilities



Source: DRA 2026.

The same liner detail and design as KPS Lined Stockpile is used. See Figure 18.56 for the AMD Stockpile liner detail.

Figure 18.56 AMD Stockpile Liner Detail



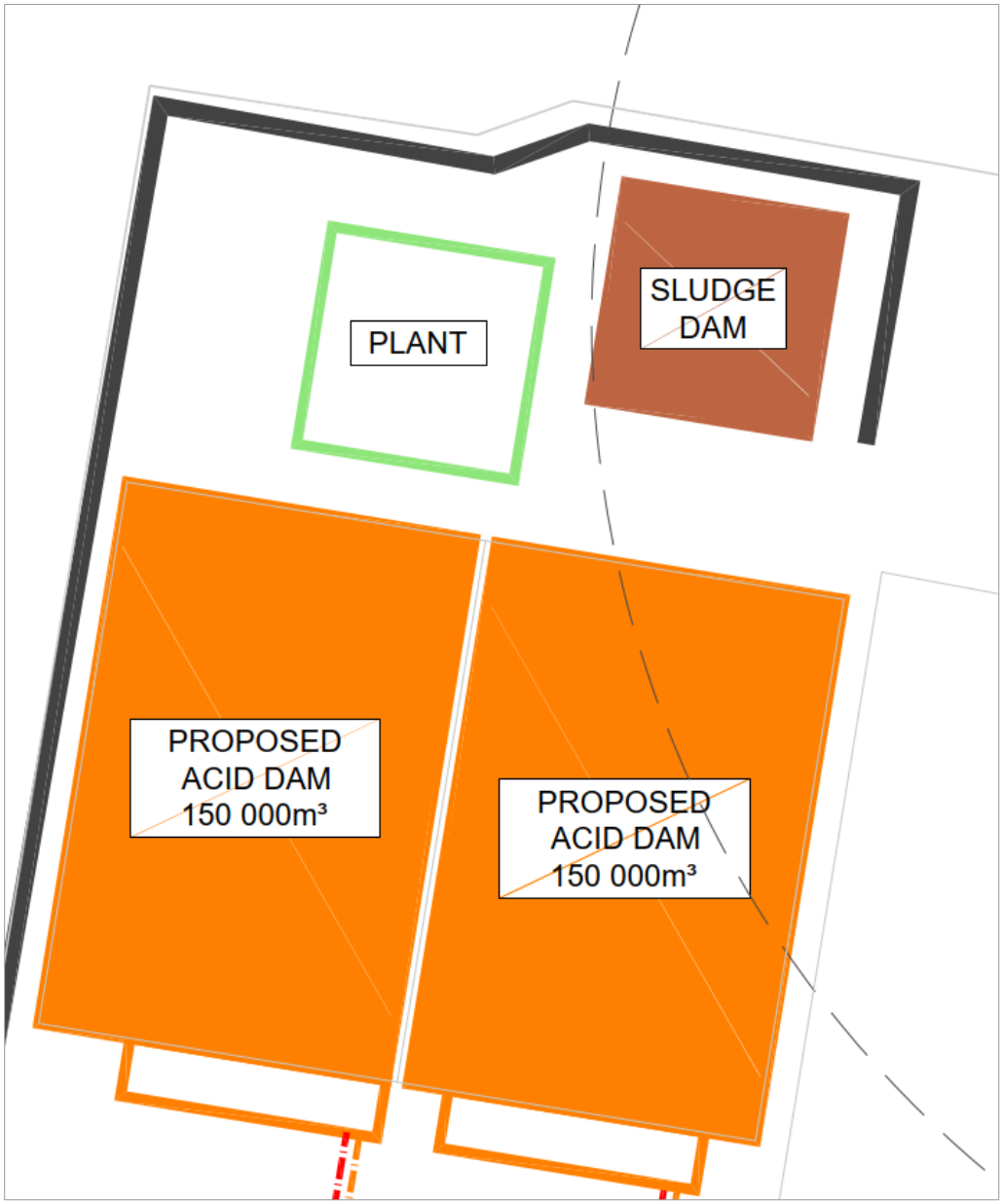
Source: DRA 2026.

18.9.3.4 Acid Water Pollution Control Dams

The Acid Water Pollution Control Dams capacities are not finalized. An updated design, based on the Waste Rock profile, as per the mine design is required. Based on a previous design done by WSP, the requirements for the 110 Ha stockpile consist of two, 150 000 m^3, HDPE Lined dams. The dam liner details used is the same as the KPS Stormwater Pollution Control Dam's design. See Figure 18.57 for the layout of the proposed acid water pollution control dams consist of two, 150 000 m^3, HDPE Lined dams. The dam liner details are the same as the KPS Stormwater Pollution Control Dam's design. See for the

The acid water pollution control dams consist of two, 150 000 m^3, HDPE Lined dams. The dam liner details are the same as the KPS Stormwater Pollution Control Dam's design.

Figure 18.57 Illustrative Acid Mine Water - Pollution Control Dams



Source: DRA 2026.

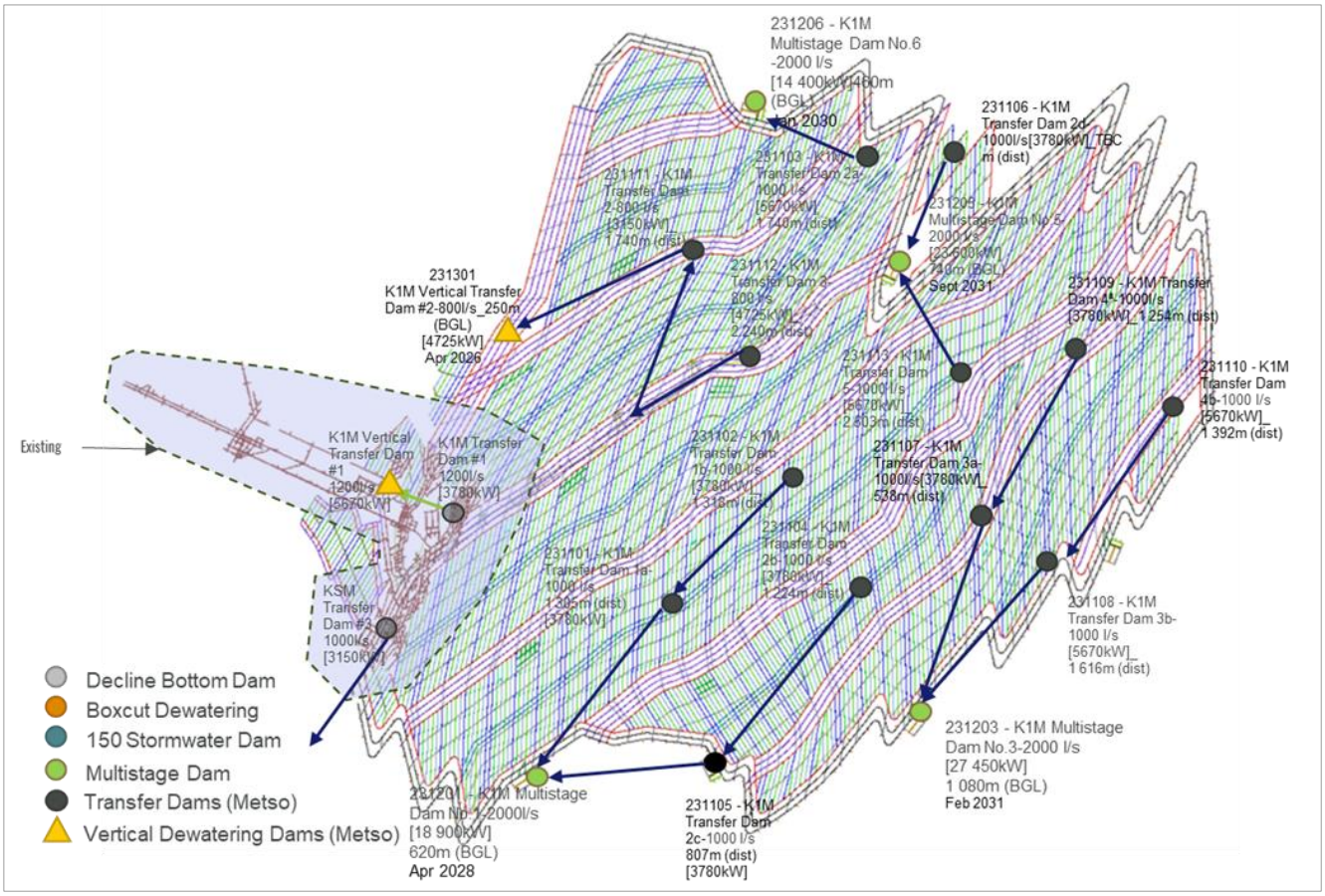
18.9.3.5 Acid Water Treatment Facility

The acid water treatment facility is a turn-key package. A terrace will be provided.

18.9.3.6 Kamoa 1 Mine Dewatering Infrastructure

The location of LoM dewatering infrastructure for Kamoa 1 mine is shown in Figure 18.58. Existing installed pumping capacity is 1,000 l/s and the estimated peak groundwater inflow is 7,700 l/s. Four multistage and one vertical transfer pumpstations with combined pumping capacity of 8,600 l/s are planned for installation over life-of-mine.

Figure 18.58 Kamoa 1 UG Mine Pumpstation Location Plan



Source: DRA 2026.

Total final installed capacity will exceed estimated peak ingress with a safety margin of 1.25 as summarized in Table 18.18.

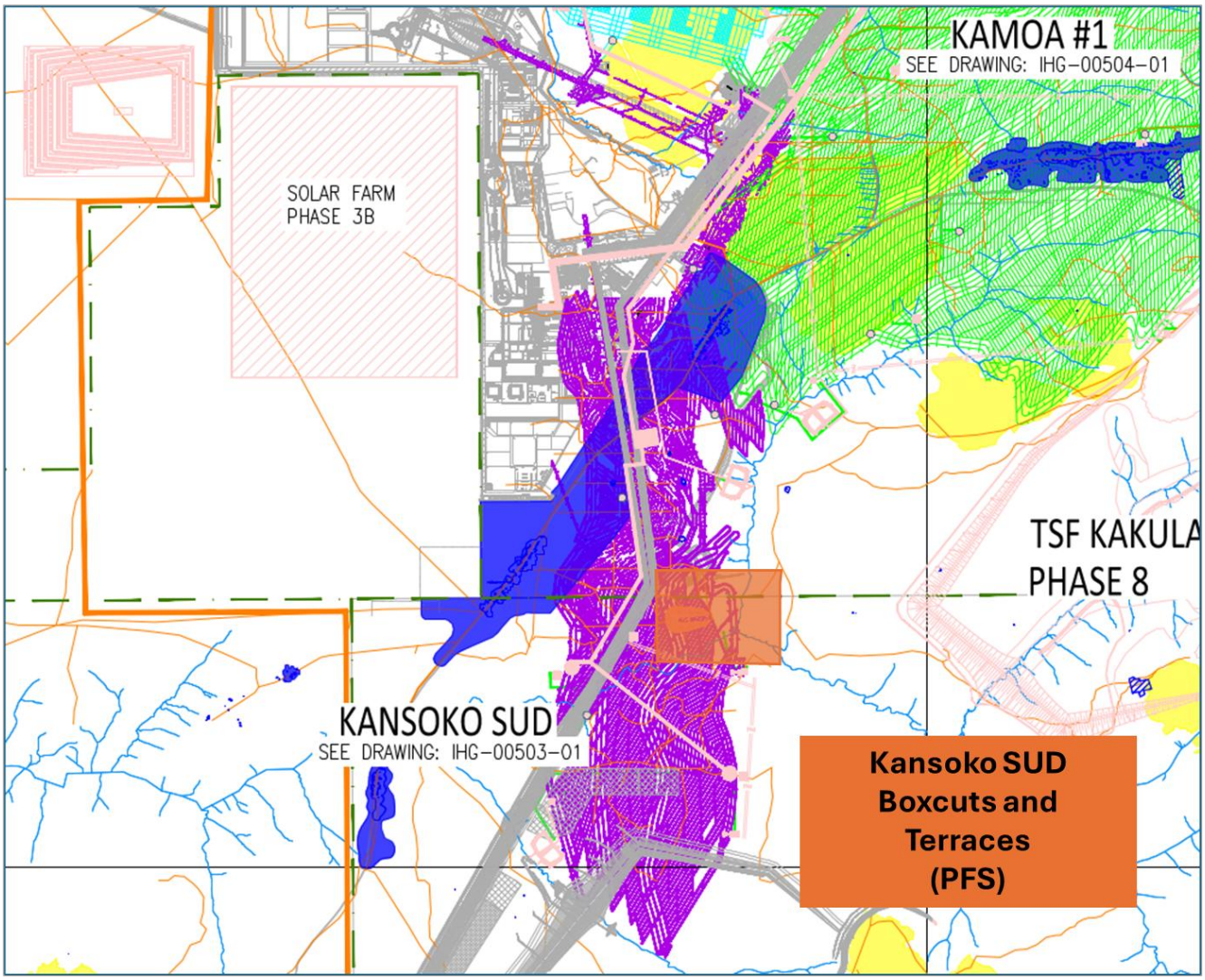
Table 18.18 Kamoa 1 UG Mine Dewatering Capacity Summary

Peak LOM Inflow (l/s)	Installed Capacity (l/s)	Additional Capacity Rqd. (l/s)	Planned Additional Capacity (for Installation) (l/s)	Margin (Safety Margin)
7,700	1,000	6,700	8,600	1.25

18.9.4 Kansoko Sud Mine

A plan showing the locations of the mines and key infrastructure for Kamoa 3 Mine is illustrated in Figure 18.59.

Figure 18.59 Kansoko SUD Mine - Key Infrastructure Locality Plan



Source: DRA 2026.

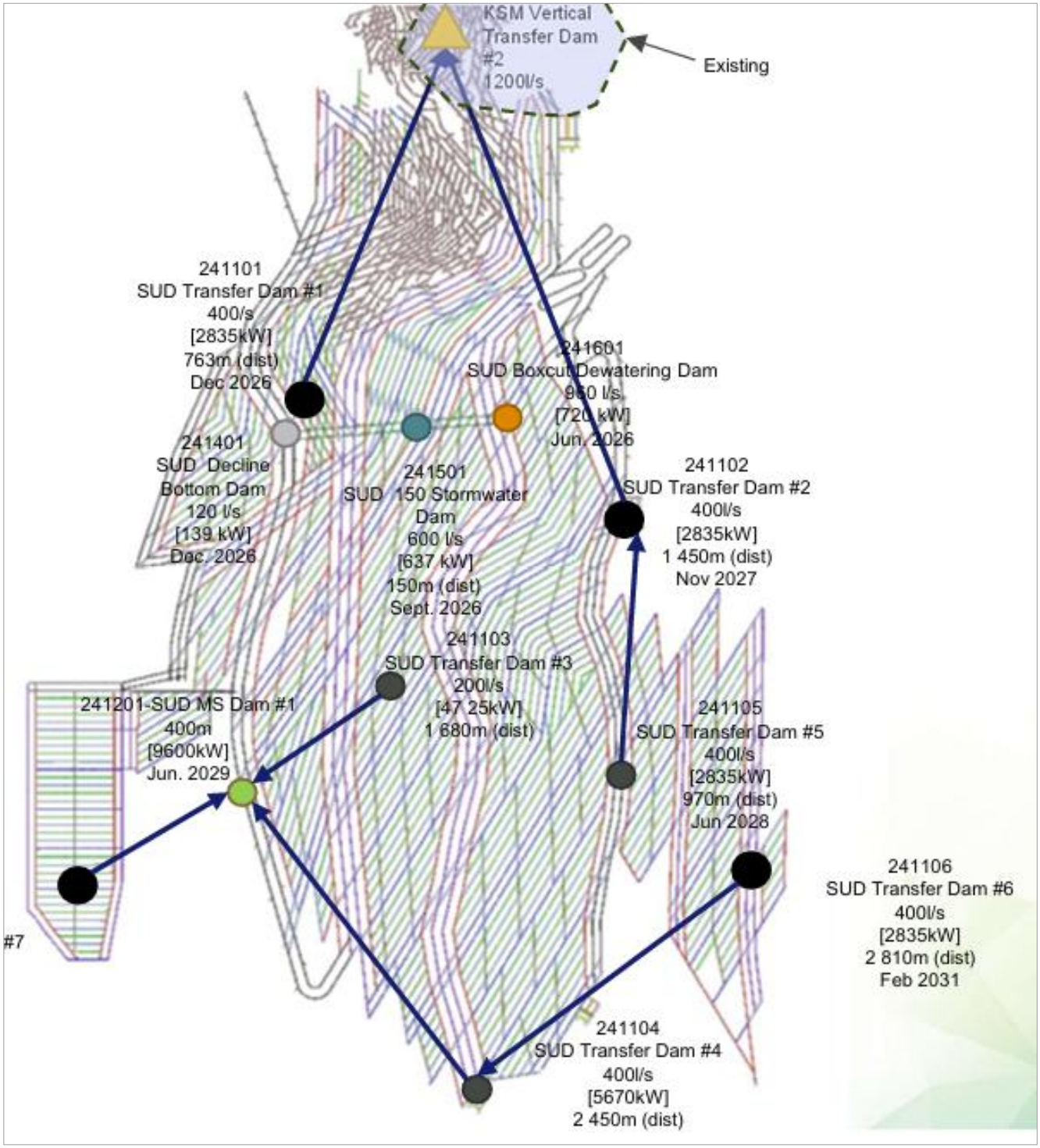
Refer to Drawing KAM-IGH-600000-00503-01 for the Surface Block Plan.

Refer to Drawing KAM-IGH-600000-00503-02 for the Underground Block Plan.

18.9.4.1 SUD Mine Dewatering Infrastructure

The location of dewatering infrastructure for SUD mine is shown in Figure 18.60. Existing installed pumping capacity is 1,000 l/s and the estimated Life-of-mine peak groundwater inflow is 900 l/s in 2037. An additional 800 l/s multistage dam will be installed due to the size of the orebody and relative location of the existing KSM Vertical Transfer Dam No.2 to the planned production areas.

Figure 18.60 SUD UG Mine Pumpstation Location Plan



Source: DRA 2026.

Total final installed capacity will exceed estimated peak ingress as summarized in Table 18.19.

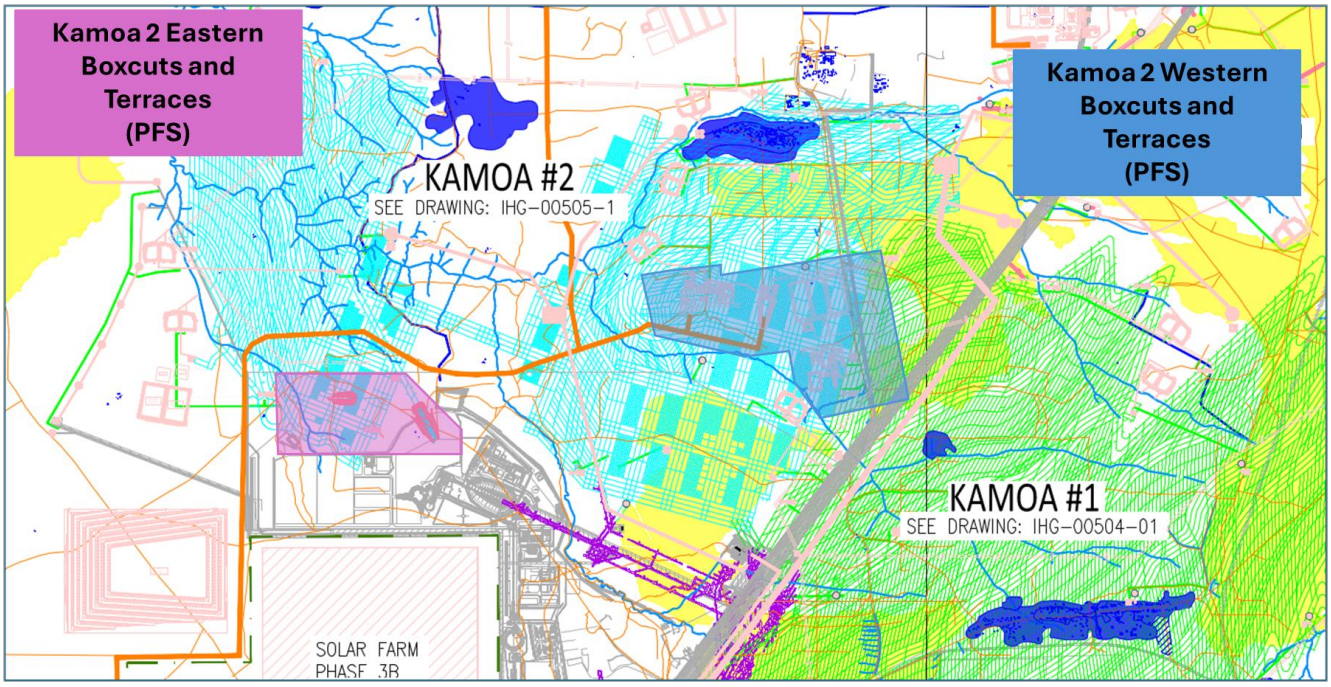
Table 18.19 SUD UG Mine Pumping Capacity Summary

Peak LOM Inflow (l/s)	Installed Capacity (l/s)	Additional Capacity Rqd. (l/s)	Planned Additional Capacity (for Installation) (l/s)	Margin (Safety Margin)
900	1,000	0,000	800	2

18.9.5 Kamoa 2 Mine

A plan showing the locations of the mines and key infrastructure for Kamoa 2 Mine is shown in Figure 18.61. The eastern mine will use the existing mining infrastructure and the eastern mine will have minor surface infrastructure.

Figure 18.61 Kamoa 2 Mine - Key Infrastructure Locality Plan

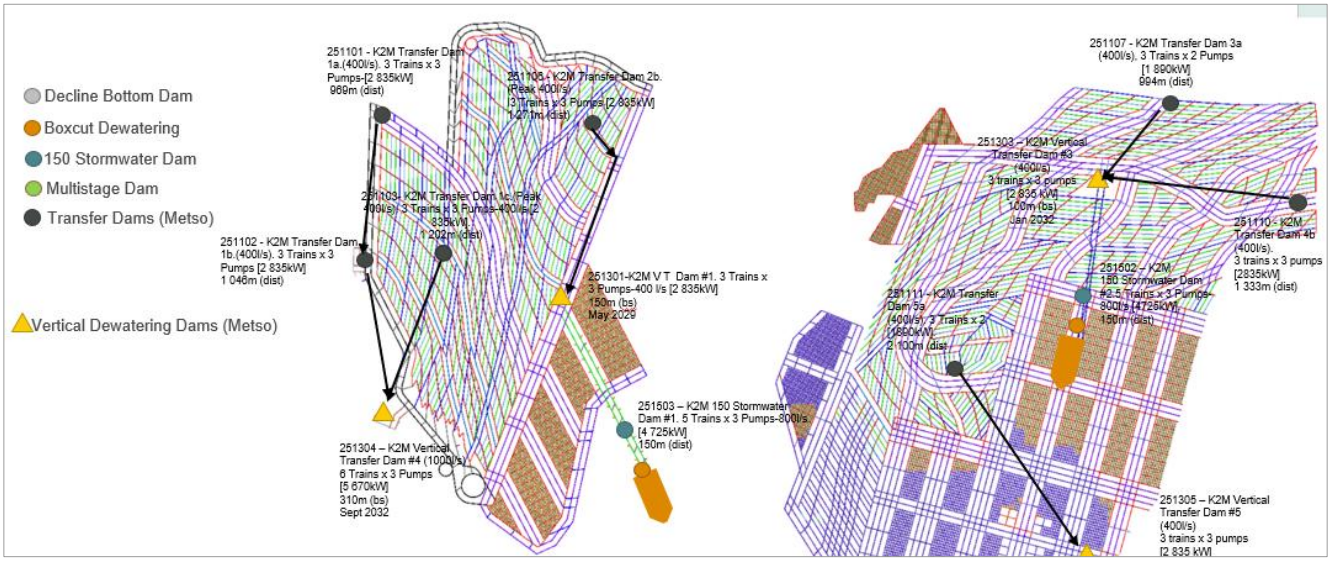


Source: DRA 2026.

18.9.5.1 Kamoa 2 Mine Dewatering Infrastructure

The location of dewatering infrastructure for Kamoa 2 mine is shown in Figure 18.62. This is a new mine with no existing installed pumping capacity and an estimated Life-of-mine peak groundwater inflow of 1,400 l/s in 2039.

Figure 18.62 Kamoa 2 UG Mine Pumpstation Location Plan



Source: DRA 2026.

Planned total final installed capacity is 2400 l/s which exceeds estimated peak ingress as summarized in Table 18.20.

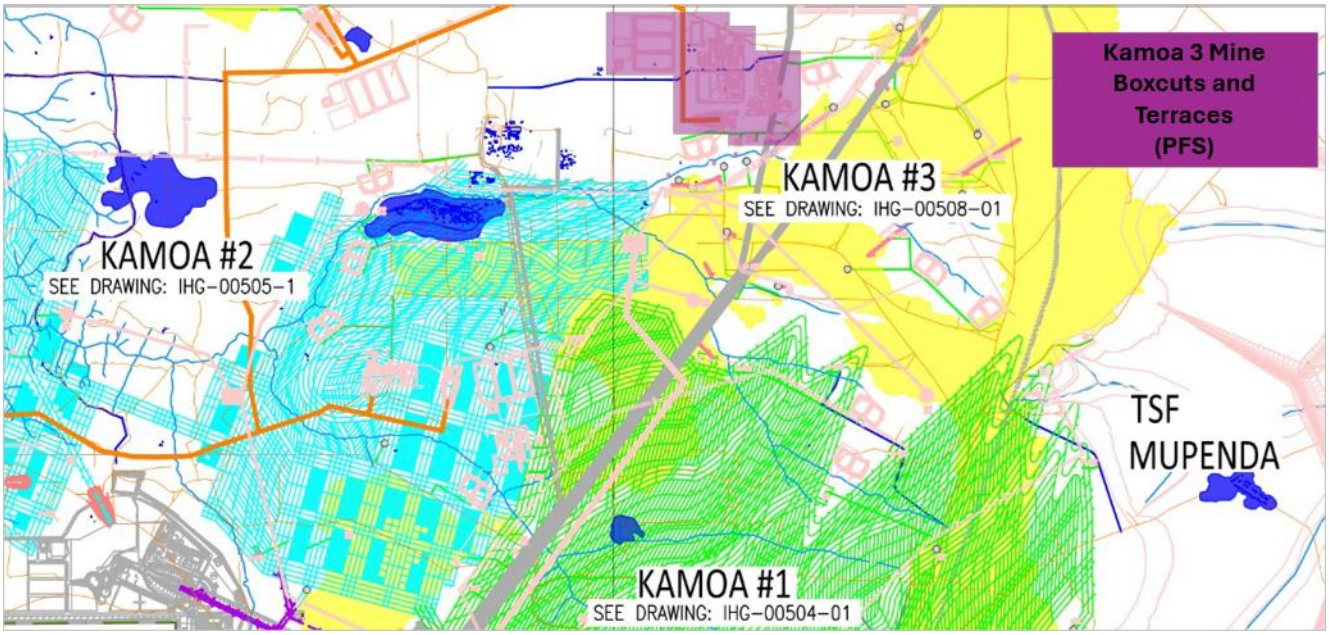
Table 18.20 Kamoa 2 UG Mine Dewatering Capacity Summary

Peak LOM Inflow (l/s)	Planned Capacity (for Installation) (l/s)	Margin (Safety Margin)
1,400	2,400	2

18.9.6 Kamoa 3 Mine

A plan showing the locations of the mines and key infrastructure for Kamoa 3 Mine is illustrated in Figure 18.63.

Figure 18.63 Kamoa 3 Mine - Key Infrastructure Locality Plan

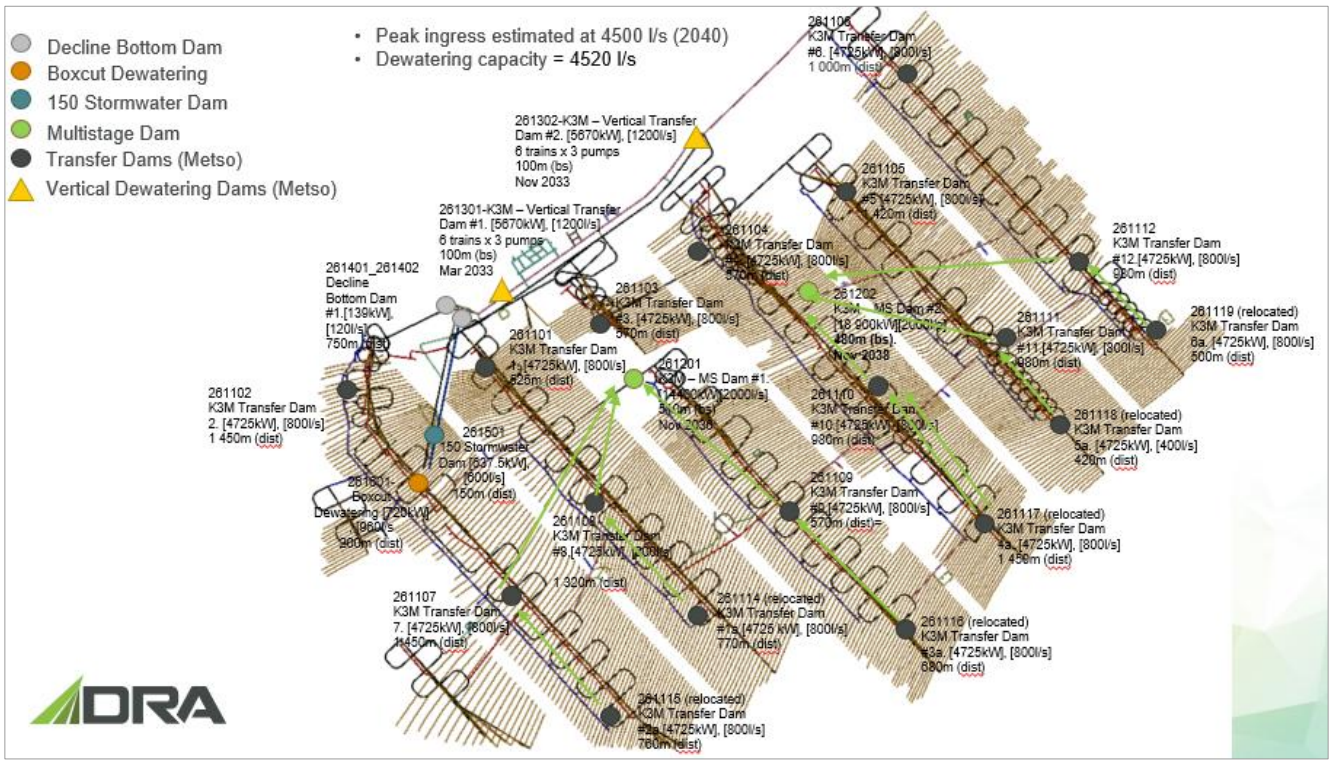


Source: DRA 2026.

18.9.6.1 Kamoa 3 Mine Dewatering Infrastructure

The location of dewatering infrastructure for Kamoa 3 mine is shown in Figure 18.64. This is a new mine with no existing installed pumping capacity and an estimated Life-of-mine peak groundwater inflow of 4,550 l/s in 2040.

Figure 18.64 Kamoa 3 UG Mine Pumpstation Location Plan



Source: DRA 2026.

Planned total final installed capacity is 6000 l/s which exceeds estimated peak ingress by approximately 32% as summarized in Table 18.21.

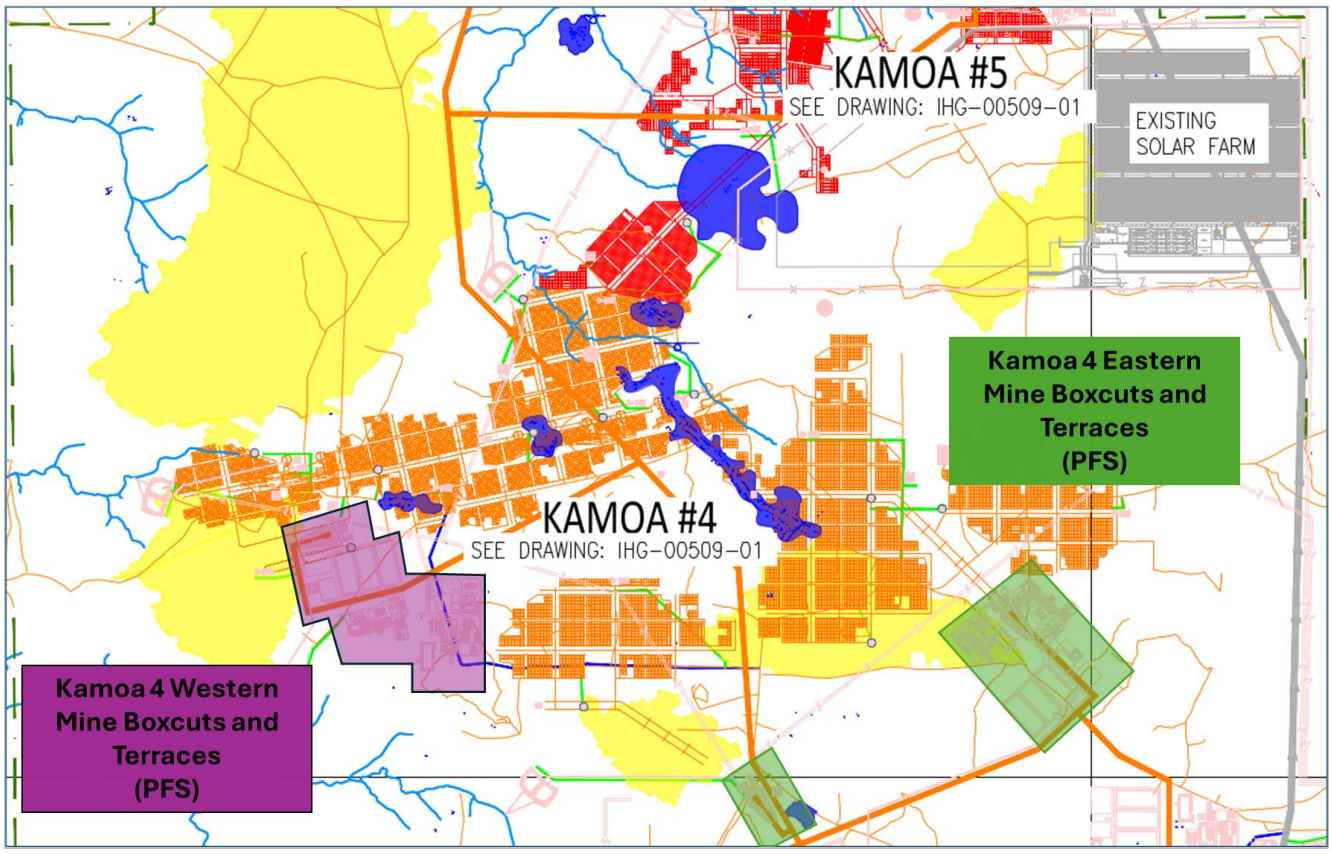
Table 18.21 Kamoa 3 UG Mine Dewatering Capacity Summary

Peak LOM Inflow (l/s)	Planned Capacity (for Installation) (l/s)	Margin (Safety Margin)
4,550	6,000	1.3

18.9.7 Kamoa 4 Mine

A plan showing the locations of the mines and key infrastructure for Kamoa 4 Mine is shown in Figure 18.65.

Figure 18.65 Kamoa 4 Mines - Key Infrastructure Locality Plan

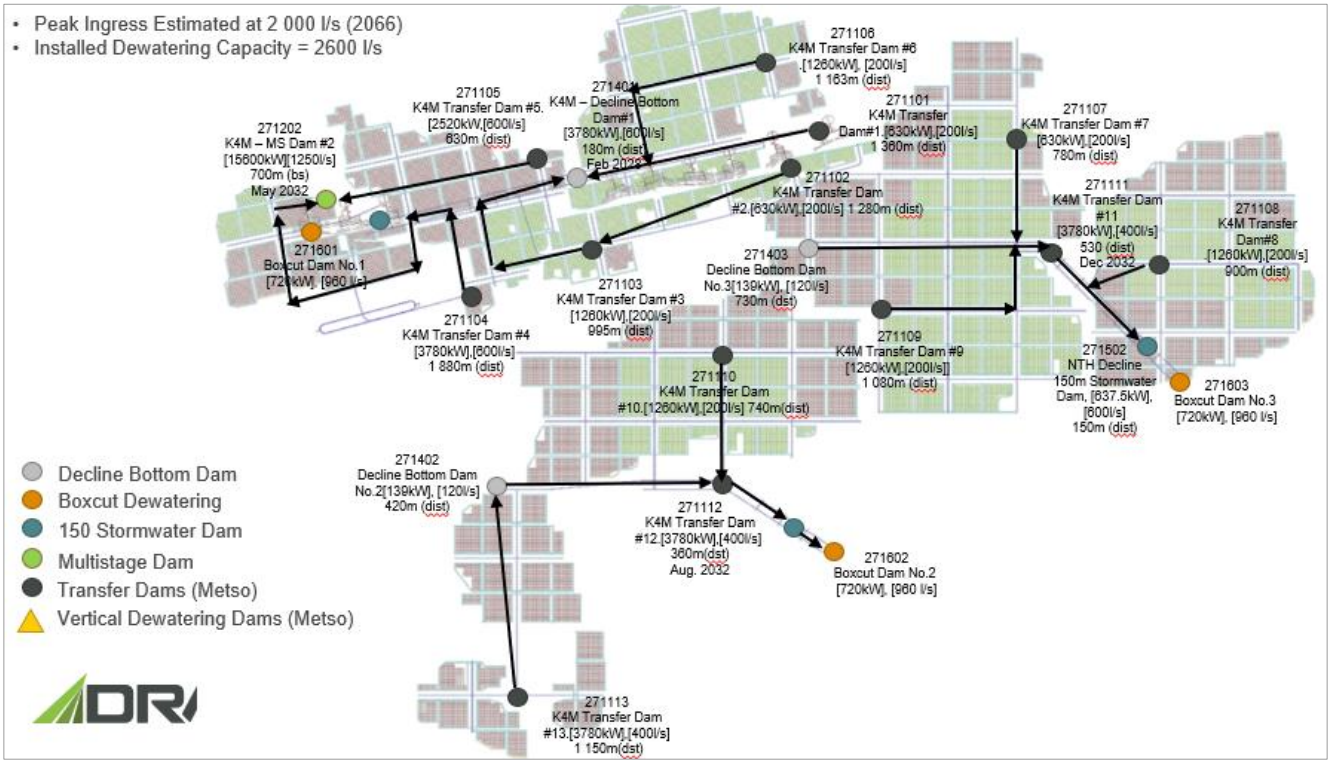


Source: DRA 2026.

18.9.7.1 Kamoa 4 Mine Dewatering Infrastructure

The location of dewatering infrastructure for Kamoa 4 mine is shown in Figure 18.66. This is a new mine with no existing installed pumping capacity and an estimated Life-of-mine peak groundwater inflow of 2,000 l/s in 2066.

Figure 18.66 Kamoa 4 UG Mine Pumpstation Location Plan



Source: DRA 2026

Planned total final installed capacity is 2600 l/s which exceeds estimated peak ingress by approximately 30% as summarized in Table 18.22.

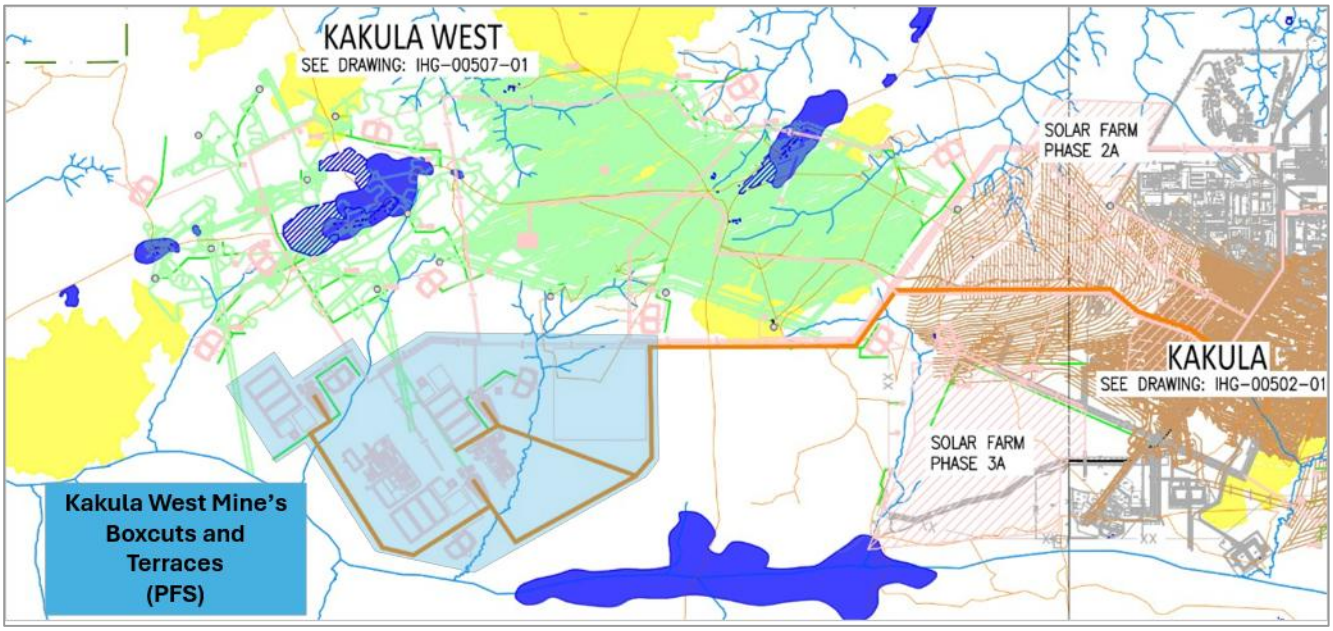
Table 18.22 Kamoa 4 UG Mine Dewatering Capacity Summary

Peak LOM Inflow (l/s)	Planned Capacity (for Installation) (l/s)	Margin (Safety Margin)
2,000	2,600	1.3

18.9.8 Kakula West Mine

A plan showing the locations of the mines and key infrastructure for Kakula West Mine is shown in Figure 18.67.

Figure 18.67 Kakula West Mine - Key Infrastructure Locality Plan

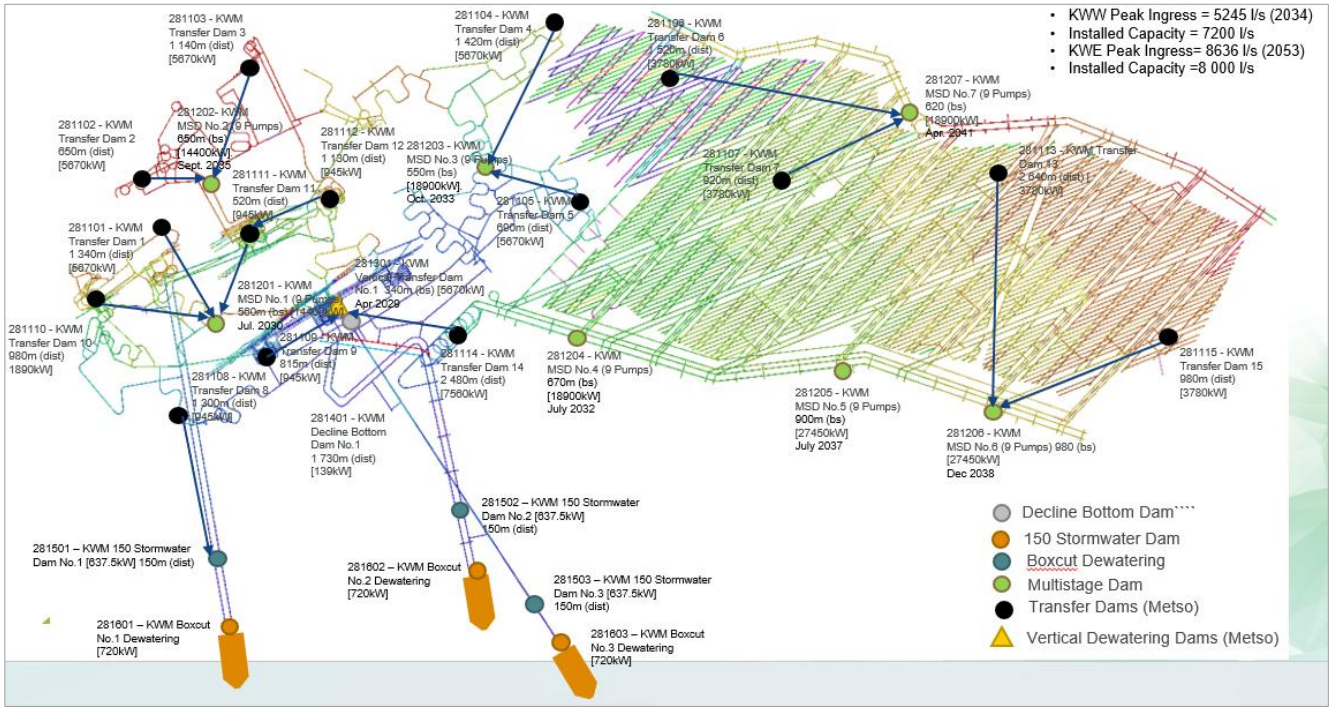


Source: DRA 2026

18.9.8.1 Kakula West Mine Dewatering Infrastructure

The location of dewatering infrastructure for Kakula West mine is shown in Figure 18.68. This is a new mine with no existing installed pumping capacity and an estimated Life-of-mine peak groundwater inflow of 13,880 l/s in 2034.

Figure 18.68 Kakula West UG Mine Pumpstation Location Plan



Source: DRA 2026

Planned total final installed capacity is 15,200 l/s which exceeds estimated peak ingress by approximately 10% as summarized in Table 18.23.

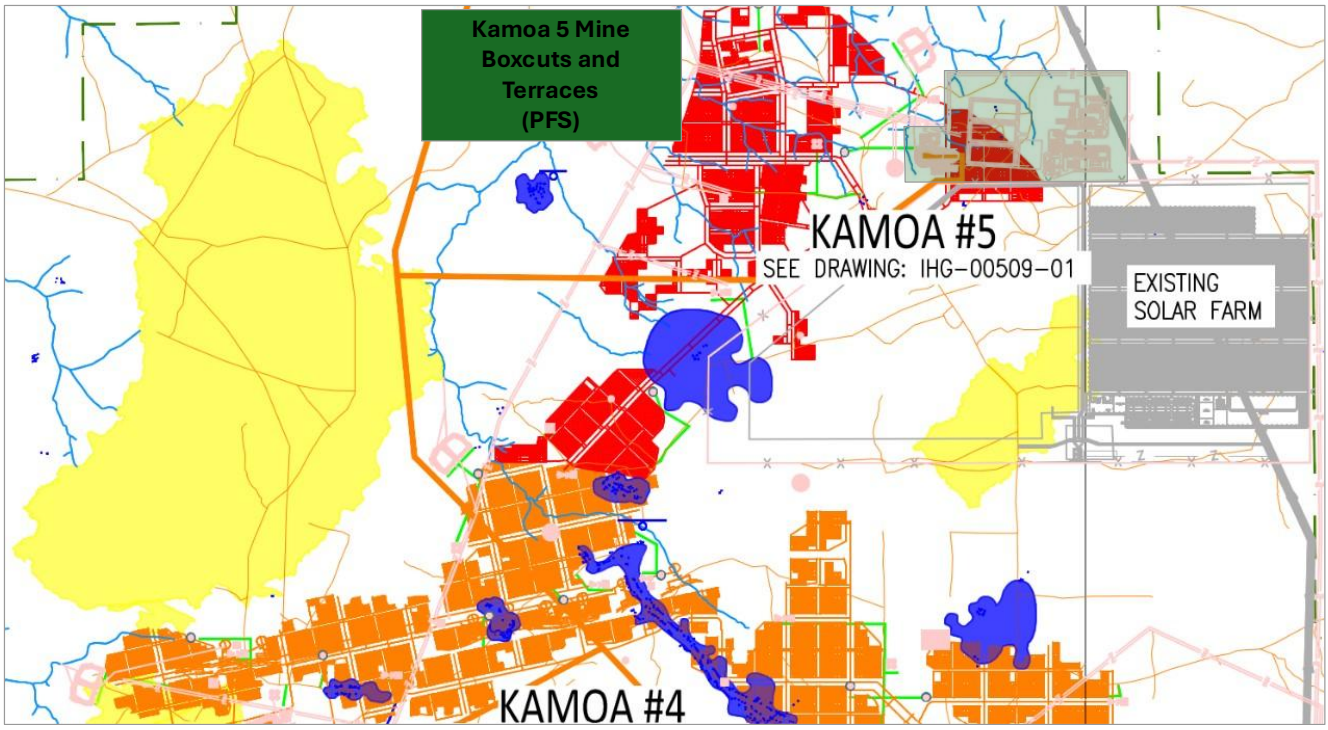
Table 18.23 Kakula West UG Mine Dewatering Capacity Summary

Peak LOM Inflow (l/s)	Planned Capacity (for Installation) (l/s)	Margin (Safety Margin)
13,880	15,200	1.1

18.9.9 Kamoa 5 Mine

A plan showing the locations of the mines and key infrastructure for Kamoa 5 Mine is shown in Figure 18.69.

Figure 18.69 Kamoa 5 Mine - Key Infrastructure Locality Plan

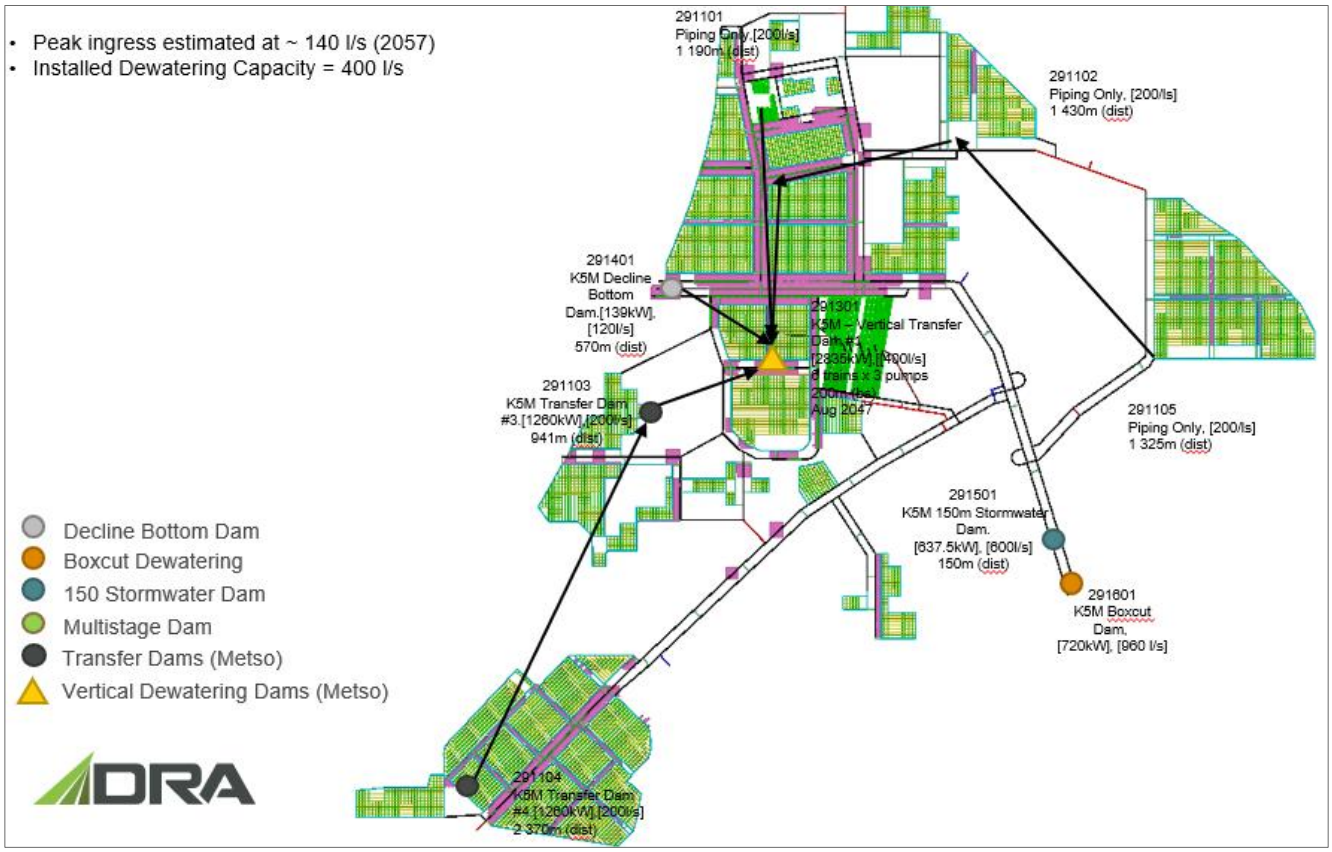


Source: DRA 2026

18.9.9.1 Kamoa 5 Mine Dewatering Infrastructure

The location of dewatering infrastructure for Kamoa 5 mine is shown in Figure 18.70. This is a new mine with no existing installed pumping capacity and an estimated Life-of-mine peak groundwater inflow of 140 l/s in 2057.

Figure 18.70 Kamoa 5 UG Mine Pumpstation Location Plan



Source: DRA 2026

Planned total final installed capacity is 400 l/s which exceeds estimated peak ingress by approximately 2.85 safety margin as summarized in Table 18.24.

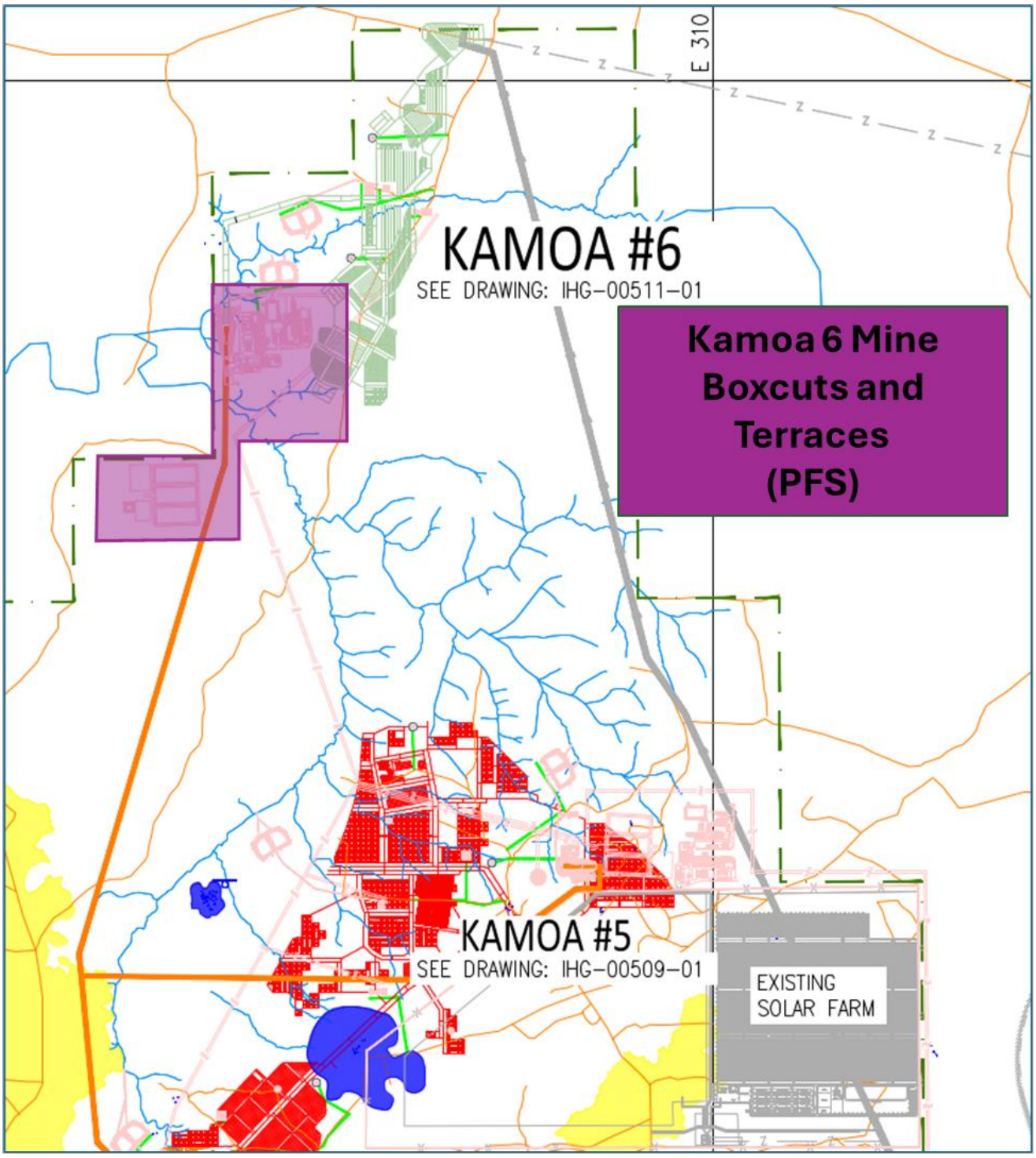
Table 18.24 Kamoa 5 UG Mine Dewatering Capacity Summary

Peak LOM Inflow (l/s)	Planned Capacity (for Installation) (l/s)	Margin (Safety Margin)
140	400	2.85

18.9.10 Kamoa 6 Mine

A plan showing the locations of the mines and key infrastructure for Kamoa 6 Mine is shown in Figure 18.71.

Figure 18.71 Kamoa 6 Mine - Key Infrastructure Locality Plan

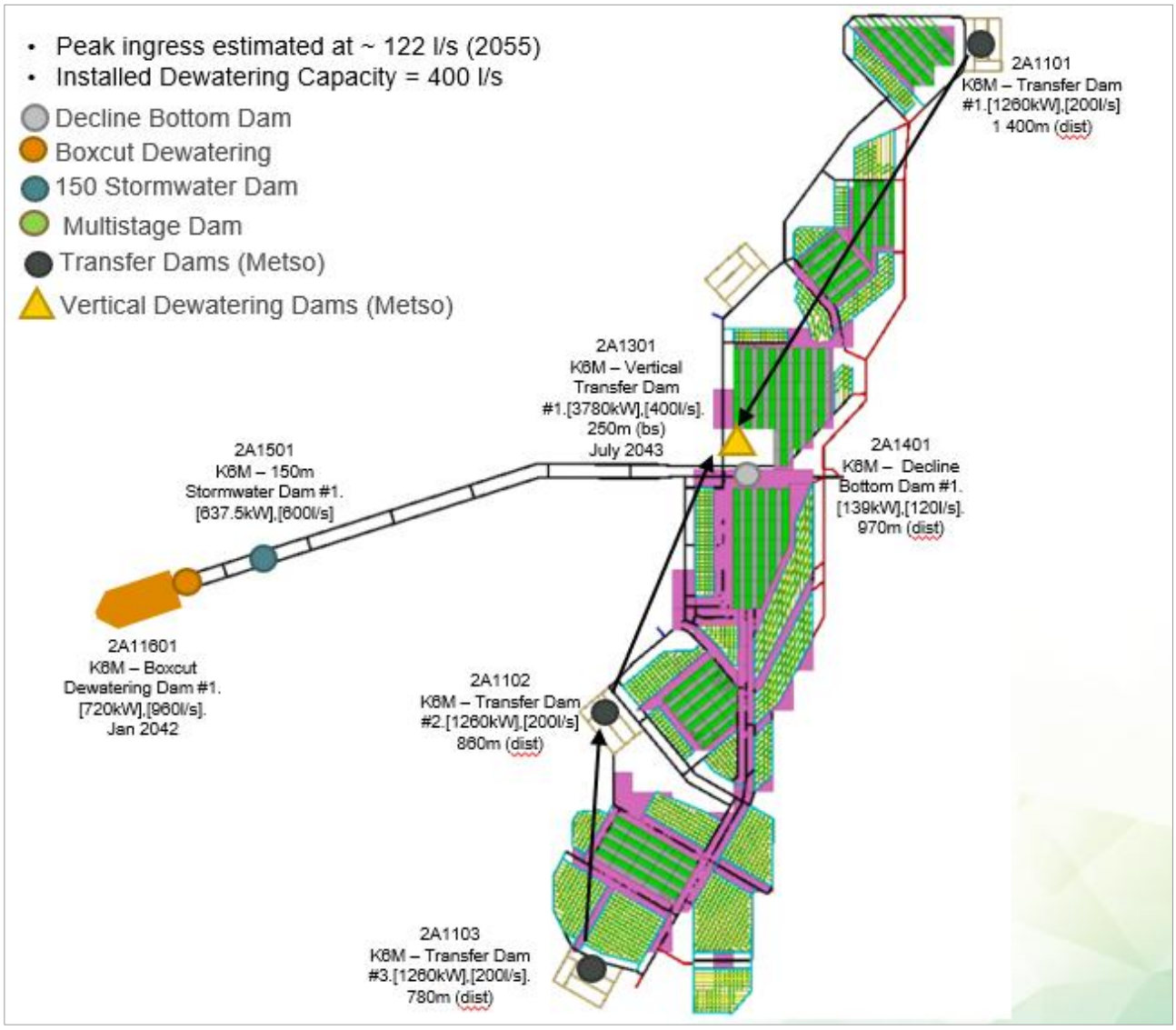


Source: DRA 2026

18.9.10.1 Kamoa 6 Mine Dewatering Infrastructure

The location of dewatering infrastructure for Kamoa 6 mine is shown in Figure 18.72. This is a new mine with no existing installed pumping capacity and an estimated Life-of-mine peak groundwater inflow of 122 l/s in 2055. Planned total final installed capacity is 400 l/s which exceeds estimated peak ingress as summarized in Table 18.25.

Figure 18.72 Kamoa 6 UG Mine Pumpstation Location Plan



Source: DRA 2026

Table 18.25 Kamoa 6 UG Mine Dewatering Capacity Summary

Peak LOM Inflow (l/s)	Planned Capacity (for Installation) (l/s)	Margin (Safety Margin)
122	400	3.28

18.10 Tailings Storage Facilities

Epoch Resources (Pty) Ltd (Epoch) completed the design of the Tailings Storage Facilities (TSFs) as part of the Kamoa-Kakula 2025 Pre-Feasibility Study (PFS). Over the next 25 year life-of-mine, approximately 336

million tonnes of tailings are expected to be generated and require storage at the TSF. To accommodate this volume, three TSF sites will be developed, the Kakula TSF, Mupenda TSF, and Site 8 TSF.

The Kakula TSF will consist of three cells (Cell 1, Cell 2, and Cell 3), while the Mupenda TSF and Site 8 will only be one Cell. All TSFs will be constructed as downstream-raised facilities, with multiple wall raises sourced from nearby borrow areas.

Construction of the Kakula TSF is currently in progress. Cell 1 has been completed up to Raise 6, while construction of Raise 1 and Raise 2 for Cells 2 and 3 is ongoing. These works are scheduled for completion in March and May 2026, respectively.

18.10.1 Design Criteria

The Kamoa-Kakula TSFs have been designed with a LOM tailings production of 414 Mt over 30-years (after backfill offtake), which includes tailings generated by the Kakula and Kamoa concentrators, as well as the smelter. The design criteria are summarized in Table 18.26.

Table 18.26 Design Criteria

Item	Criteria	Value	Source
1	Design life of facility	25-years	Kamoa
2	Total Tailings	336 Mt	Kamoa
2.1	Kakula	220.7 Mt	Kamoa
2.2	Kamoa	100.4 Mt	Kamoa
2.3	Smelter Tailings	15.6 Mt	Kamoa
3	Particle Specific Gravity	2.85	Kamoa
4	Average Dry Density	1.32 t/m3	Epoch
5	Average Particle Size Distribution*	80% Passing 66 µm	ST Lab
6	% Solids to Water (by Mass)	48	DRA/Kamoa
7	Delivery Method	Hydraulically pumped	Kamoa
8	Geochemistry	Leachable Mine Waste	Golder

Note: *Project 95 (Kakula) is expected to generate a finer PSD

18.10.2 TSF Site Selection

A site selection study was initially conducted in 2014 for the Kamoa plant. Two sites were shortlisted, Mupenda and Site 8. Mupenda was selected as the preferred option due to the low consequence classification relative to Site 8.

A subsequent study was undertaken to identify suitable sites for storing tailings from Kakula. The Kakula site was identified as the preferred location.

Following a re-evaluation in 2025, incorporating additional test work and updated production forecasts, the total tailings storage requirement over the life of the mine was determined to be approximately 336 Mt. Reassessments showed that the combined capacity of the initially planned Kakula TSF and Mupenda TSF would be insufficient. Site 8 was therefore included in the study.

18.10.3 Geotechnical Investigation

A geotechnical investigation of the Kakula TSF Cell 1 site was undertaken by Knight Piésold, while investigations for Cells 2 and 3 were performed by Bear Geo Consultants (BGC). The investigations included test pit and borehole profiling, soil sampling, and laboratory testing. The primary objectives were to determine geotechnical parameters, soil horizons, in-situ depths, and to identify any problematic soils that could affect stability or seepage.

In general, the soils encountered across the area comprise transported materials, predominantly aeolian and alluvial deposits, underlain by residual diamictite. Laterite and residual diamictite are more prevalent within valley areas. In certain locations within the containment wall footprints, deeper horizons of sandy alluvial and aeolian soils were encountered, necessitating deeper foundation excavations. Similarly, soil profiles along ridges and upper-slope areas indicate thicker sequences of transported material, primarily alluvial and/or aeolian in origin

Due to the highly variable soils, strict quality control of borrow materials will be required during construction to ensure that fill within the containment wall and box-cuts meets the design specifications.

Groundwater conditions varied across the three cells. In Cell 1, groundwater was encountered in several test pits at varying depths during the investigation conducted from 24 November 2018 to 22 April 2019, coinciding with the regional wet season. Cell 2 was investigated from April to June 2024, at the end of the wet season, during which groundwater was observed at an average depth of 4 m. Higher water tables were noted in valley areas and zones underlain by Aeolian sands, and CPTu probing confirmed these observations. Cell 3 was investigated in October 2024, during the dry season, and groundwater was encountered at an average depth of 3.1 m. The site's location within a valley and along the lower reaches of several drainage lines contributes to relatively high water table conditions throughout the area.

Sparse test pitting at Mupenda has provide some information on the soils at the site, however additional investigative work is planned at Mupenda and Site 8.

18.10.4 Seepage and Stability Assessment

Seepage and stability analyses were undertaken as part of the designs of the Kakula TSF Cells 1, 2, and 3; under both drained and undrained conditions. Soil parameters were based on laboratory and in situ (CPTu) testing. Seepage analyses evaluated phreatic surface development and the effectiveness of drainage systems in reducing pore water pressures, while slope stability analyses assessed containment wall performance under various loading conditions in accordance with ICOLD guidelines and DRC regulatory requirements. As ICOLD does not specify minimum pseudo-static factors of safety, a simplified seismic deformation analysis using the method of Bray and Travasarou (2007) was also performed.

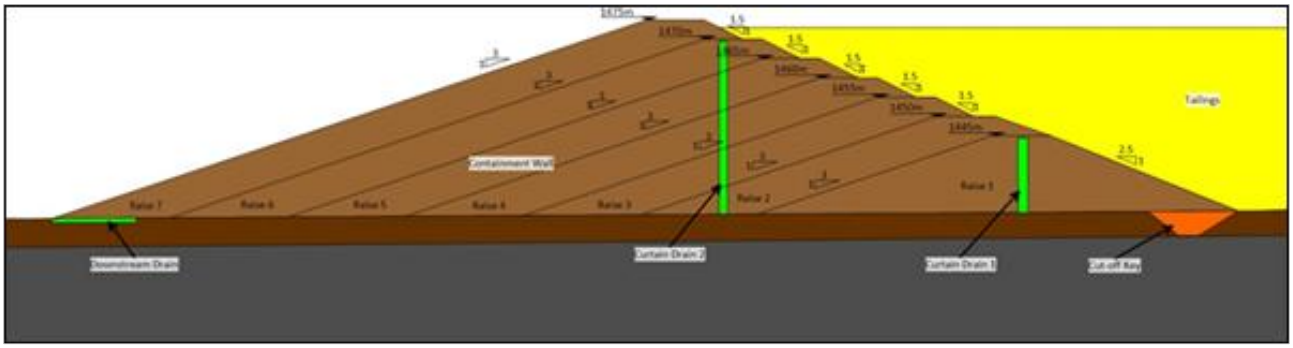
The results confirm that the facility is stable under both static and pseudo-static loading conditions, with factors of safety exceeding 1.5 for static and 1.1 for pseudo-static cases.

18.10.5 Operational and design philosophy

The depositional technique selected for this project was a valley containment, hydraulically deposited, spigot facility. An engineered earthfill wall is to be constructed utilizing nearby sources of soil. Incremental raising of the wall is to be undertaken in a downstream direction, providing the necessary strength and robustness that the tailings product lacks.

The Kamoa–Kakula Project will include three TSFs, the Kakula, Mupenda, and Site 8 facilities. Each TSF will be developed in phases and raised progressively as a downstream-type facility in 5 m vertical increments (Figure 18.73).

Figure 18.73 Typical Phasing of the TSF Embankment



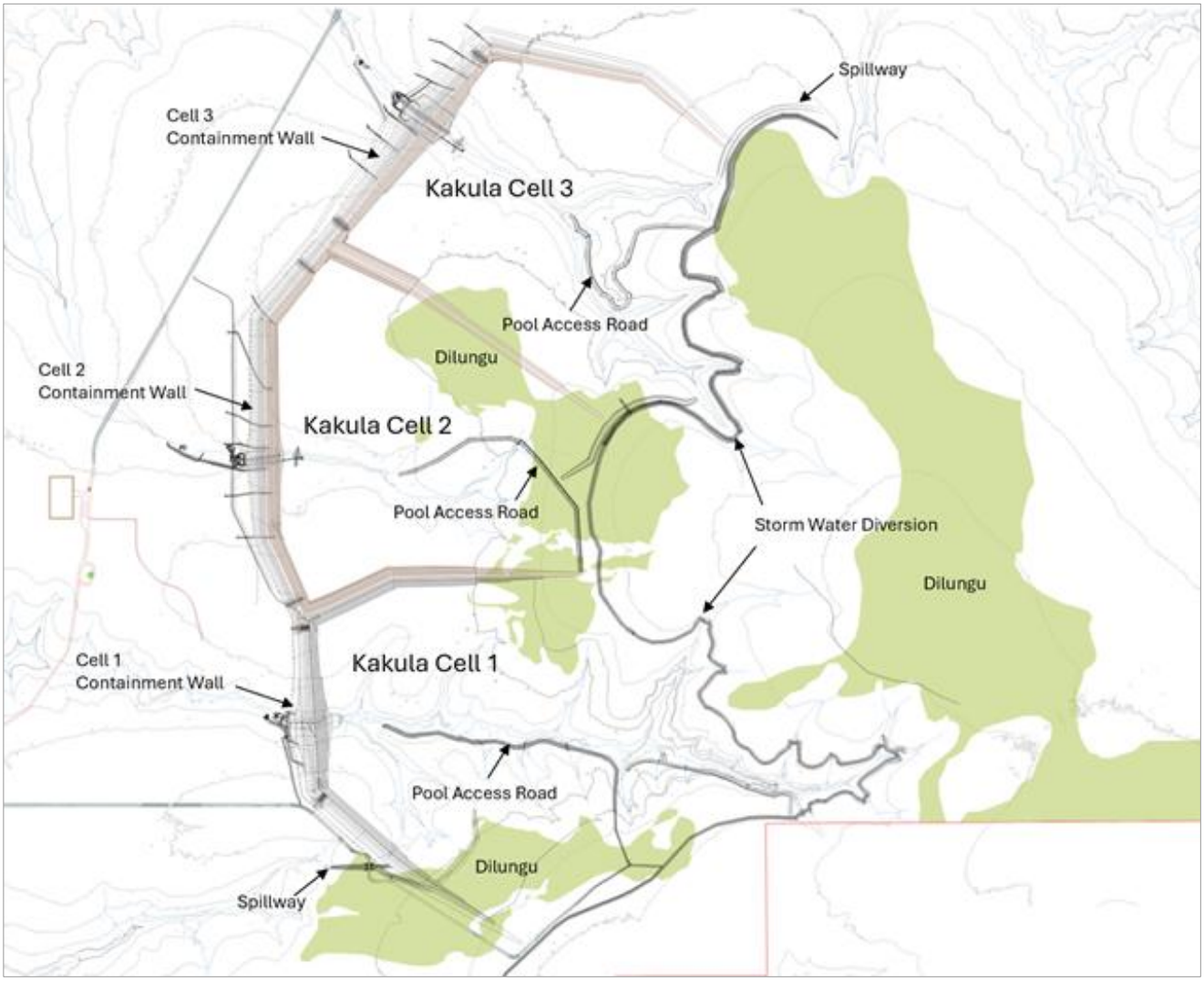
Source: Epoch 2025

18.10.6 Key Design Features

The layouts of the respective TSFs are shown in Figure 18.74 to Figure 18.76 and the key design features of the facility are as follows:

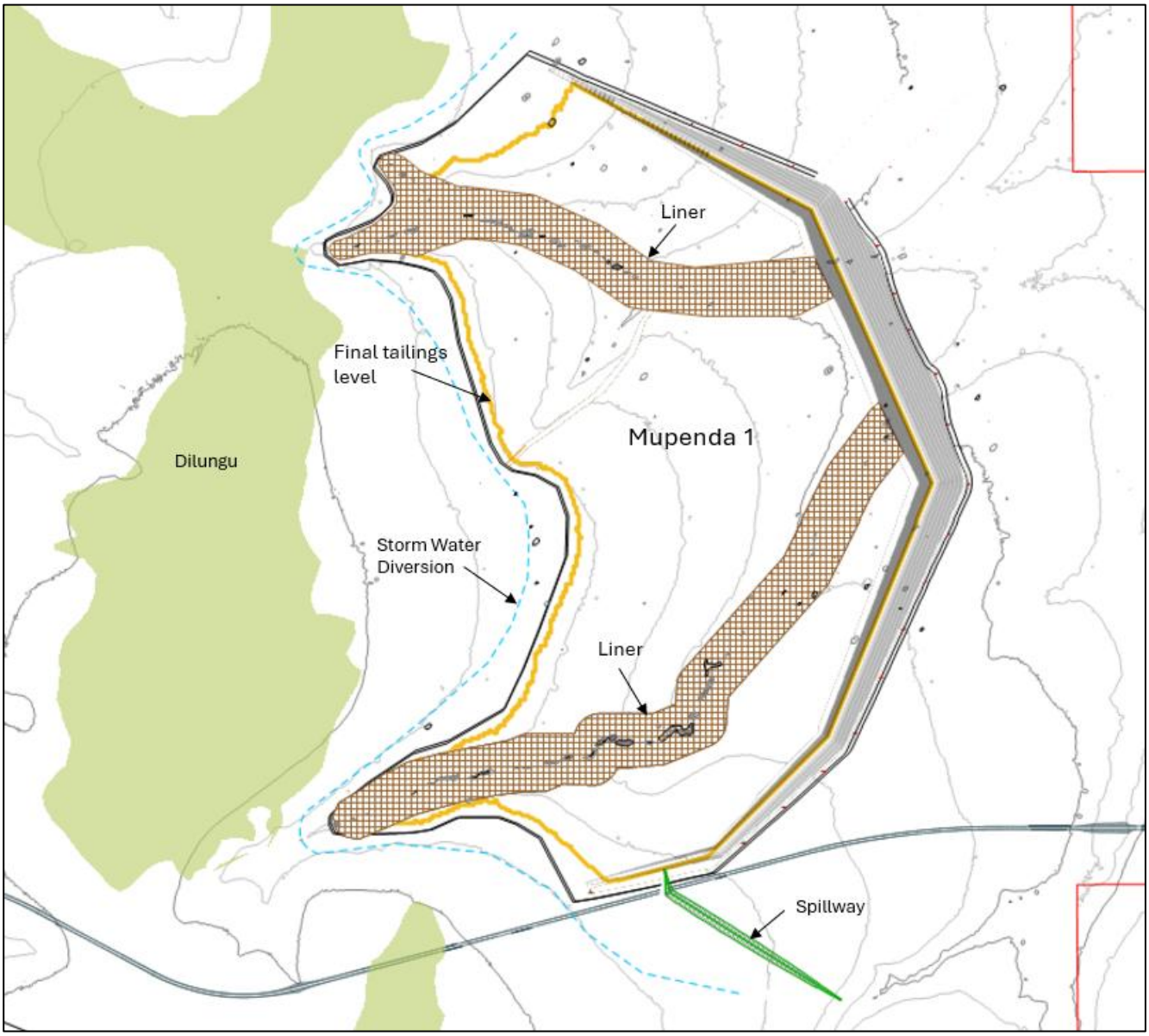
- Full containment, downstream construction method, with open ended deposition.
- An engineered, earth fill embankment with an HDPE liner on the upstream slope.
- Pool access road.
- A floating decant system.
- Curtain drain seepage collection systems inside the containment wall (to control the phreatic level within the wall).
- An HDPE liner (over highly permeable soils).
- Storm water diversion trenches.
- Emergency spillways.

Figure 18.74 Kakula TSF Layout



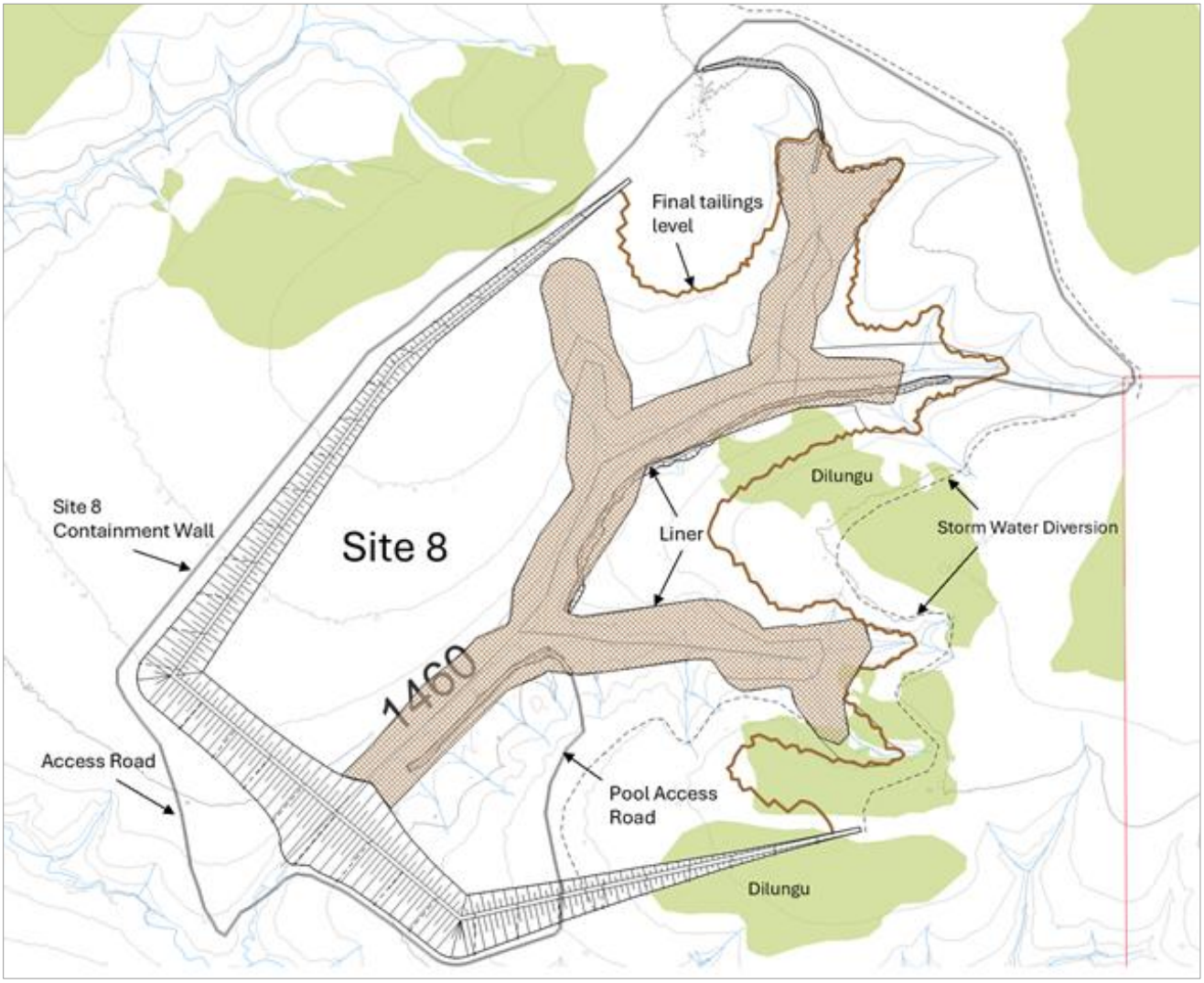
Source: Epoch 2025

Figure 18.75 Mupenda TSF Layout



Source: Epoch 2025

Figure 18.76 Site 8 TSF Layout



Source: Epoch 2025

18.10.7 TSF development strategy

The TSF construction schedule has been developed to account for the anticipated construction duration of each wall raise, the incremental storage capacity gained per raise, and a 12-month contingency allowance to mitigate potential delays. This approach ensures the continuous availability of tailings storage capacity throughout the life-of-mine.

Table 18.27 summarizes the construction schedule and storage capacity provided by each TSF, while Figure 18.77 illustrates the planned construction sequence for the facilities. Table 18.28 presents the tailings deposition schedule for each TSF.

Kakula TSF Cell 1 has been built to Raise 6 providing a total of 66Mt of tailing storage capacity. Currently Cell 1 receives tailings from both Kakula and Kamoa. Construction of Cells 2 and 3 is currently underway and is expected to be completed in March 2026 and May 2026, respectively. Cell 1 Raise 7 has been delayed deferring costs due to the higher volume of earthworks associated with the final Raise.

Tailings deposition will be split between Cell 2 and 3, such that Kakula tailings will be directed to Cell 2, while tailings from Kamoa will be directed to Cell 3.

After Cell 1, 2 and 3 have reached capacity, tailings will be deposited at Site 8. Construction of Site 8 TSF will need to commence six months after completion of the Kakula Cell 1 Raise 7.

Construction of Mupenda TSF will start 24 months before Site 8 reaches capacity to ensure that construction delays do not affect mining operations.

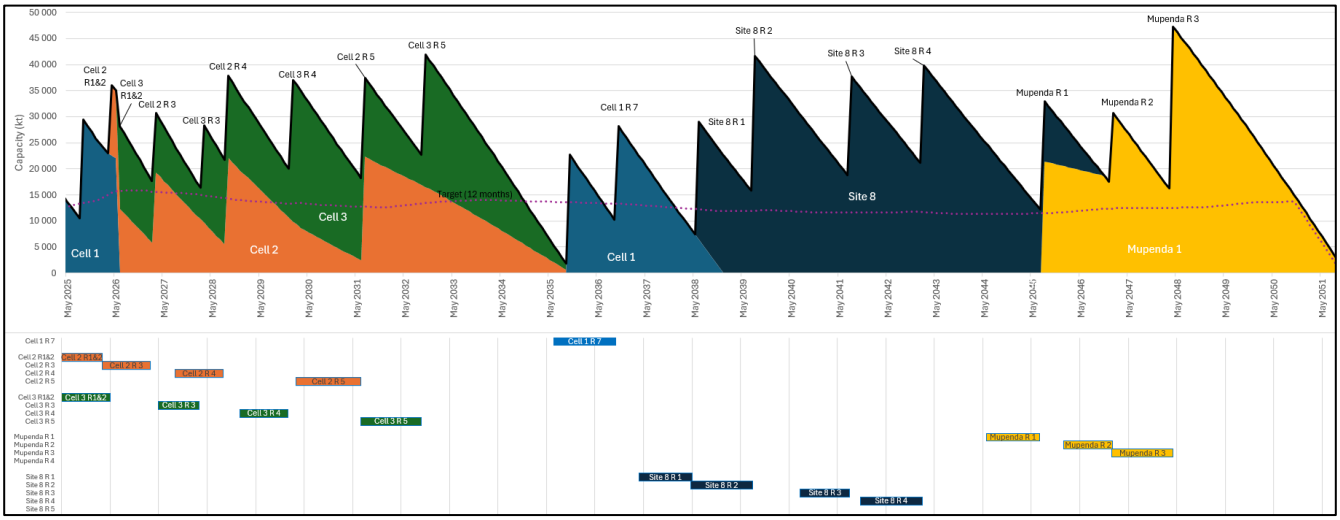
Table 18.27 TSF construction schedule

TSF	Construction Start	Construction End	Capacity (Mt)	Cumulative Capacity (Mt)
Cell 2 R1&2	31/03/2025	31/03/2026	14	66
Cell 2 R 3	31/03/2026	31/03/2027	14.3	
Cell 2 R 4	31/12/2027	31/12/2028	17.3	
Cell 2 R 5	02/12/2030	31/03/2032	20.4	
Cell 3 R 1&2	26/04/2025	31/05/2026	16.4	67.8
Cell 3 R 3	30/05/2027	31/03/2028	13.1	
Cell 3 R 4	31/01/2029	31/01/2030	18.	
Cell 3 R 5	31/07/2031	31/10/2032	20.2	
Cell 1 R 7	31/10/2036	31/01/2038	19.2	84.1
Mupenda 1 R 1	27/07/2038	31/08/2039	21.3	99.8
Mupenda 1 R 2	31/05/2040	31/05/2041	14.3	
Mupenda 1 R 3	31/07/2041	31/10/2042	23.3	
Mupenda 1 R 4	30/08/2043	28/02/2045	40.9	
Mupenda 2 R 1	28/10/2047	30/11/2048	18.8	50.4
Mupenda 2 R 2	21/03/2049	30/06/2050	8.8	
Mupenda 2 R 3	24/11/2051	30/11/2052	10.5	
Mupenda 2 R 4	20/12/2052	31/03/2054	12.3	
Site 8 R1	08/11/2042	30/06/2044	22.6	88.8
Site 8 R2	07/10/2045	31/12/2046	26.8	
Site 8 R3	05/02/2047	30/04/2048	19.8	
Site 8 R4	04/02/2050	30/04/2051	19.6	

Table 18.28 TSF Deposition schedule

TSF	Deposition Start	Deposition End
Cell 2	30/04/2026	30/09/2036
Cell 3	30/06/2026	31/08/2036
Cell 1	31/08/2036	30/09/2040
Mupenda 1	30/09/2040	31/05/2049
Mupenda 2	31/01/2045	28/02/2055
Site 8	31/05/2049	31/12/2055

Figure 18.77 TSF Capacity availability and Construction Scheduling



Source: Epoch 2025

18.10.8 Risk Identification

According to the GISTM consequence classification, the Kakula TSF was classified as “High”.

No formal breach assessment has been completed for Mupenda and Site 8, however Mupenda is expected to be “Low” to “Significant”, while Site 8 is expected to be “High” to “Very High”. This is based on desktop assessments of the downstream environment and communities.

The major risks associated with the TSFs are as follows:

- Inadequate compaction of the containment wall could result in defects such as ratholing or piping, potentially leading to embankment failure. All compaction must therefore comply strictly with design specifications.
- Based on a failure mode assessment, the primary credible failure mode is a deep-seated slip failure, therefore foundation preparation, compaction control and seepage controls, comprise critical controls for the integrity of the TSFs.
- Due to the timeframes required to construction raises, strict deposition monitoring and construction scheduling is required to maintain deposition space.
- Geotechnical data at TSFs beyond the Kakula is not yet defined, which may affect feasibility or costs.

18.11 Cemented Backfill Plants

Cemented backfill plays a critical role in maintaining underground mine stability and supporting ongoing mining operations. Two primary cemented backfill types are utilized: Cemented Paste Backfill (CPB) and Cemented Aggregate Fill (CAF), each selected based on orebody requirements and operational conditions.

CPB is the primary backfilling method for most underground workings. It is produced by combining filtered tailings, water, and binder to form a pumpable paste. Once cured, CPB provides the necessary structural support to minedout stopes, ensuring stability and continuity of the mining cycle.

CAF serves as the backfilling method for more remote orebodies where annual backfill volumes are lower. CAF is produced by mixing crushed wasterock aggregate with water and binder to create a durable fill material. As with CPB, cured CAF reinforces the minedout areas and maintains local structural integrity.

18.11.1 Existing Backfill Plants

The Kamoa-Kakula complex has two operating backfill plants. The Kakula Backfill Plant consists of three modules and services the Kakula orebody. The Kamoa 1 Backfill Plant consisting of two modules currently services the Kansoko and Kamoa 1 orebody.

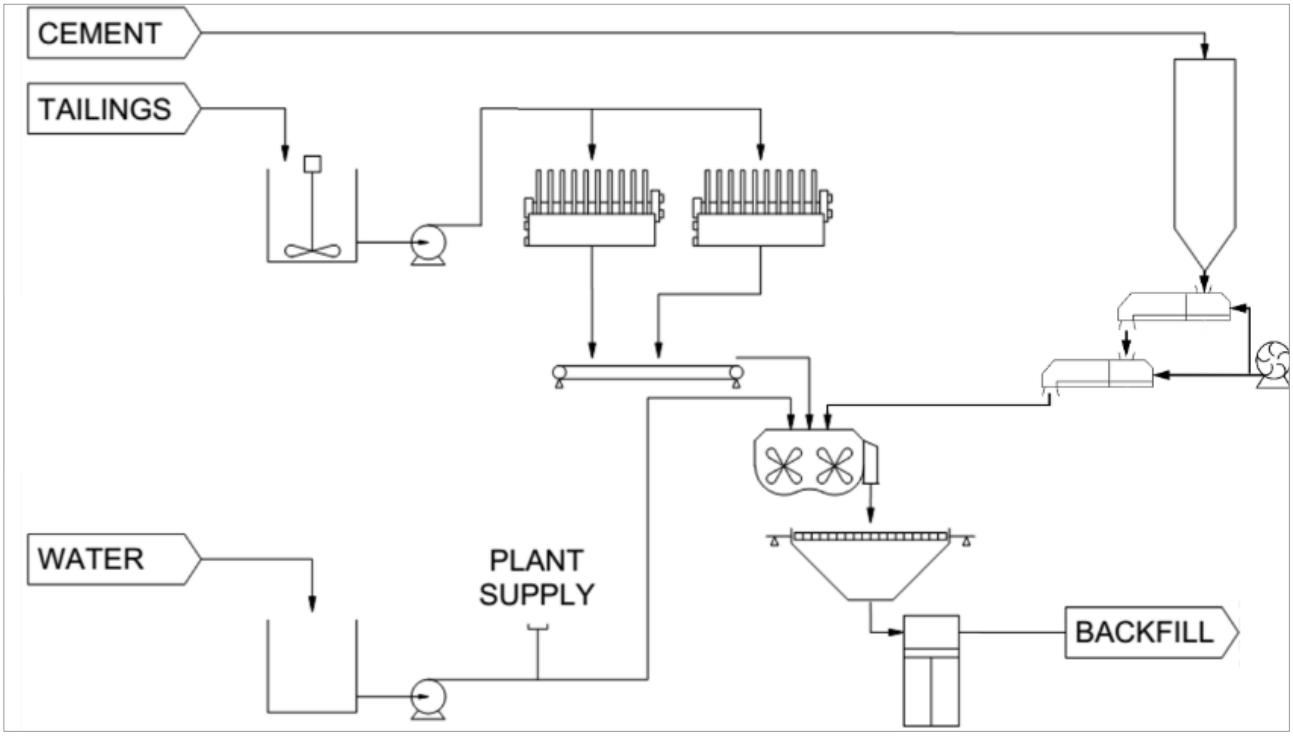
18.11.1.1 Kakula Backfill Plant

The Kakula plant (Figure 18.78) is supplied with tailings from the Kakula concentrator plant. Tailings are pumped from the tailings thickener to the backfill plant tailings storage tanks (one per module) through overland pipelines. The tailings tanks serve as a buffer to short-term tailings supply outages. Tailings are pumped to vacuum disc filters (two per module) for local dewatering. The disc filters discharge filter cake onto a filter cake conveyor which dumps into a continuous paste mixer. Trim slurry, water and binder are added to the mixer to create a homogenous CPB.

The binder system, consisting of two, 2000-tonne storage silos is supplied directly from binder delivery trucks. Binder is transferred from the storage silos to either one of the three 750-tonne dosing silos (one per module) through pneumatic conveying. The dosing system consists of a blower, impact weigher and air slides which feeds binder to the continuous mixer. A binder system upgrade is currently underway at to replace the current system with a weigh belt and screw conveyor to increase the system's throughput capacity. The upgrade was necessary to account for higher binder dosing rates in future due to a finer tailings PSD.

The paste in the continuous mixer discharges by overflowing into a paste hopper. The hopper facilitates continuous flow of paste to the underground. Paste from the hopper reports to a 150-bar rated hydraulic piston pump, which distributes the paste through a DN 200 fill reticulation system to the mined stopes. The reticulation system consists of overland paste pipelines, surface to underground paste boreholes and underground reticulation piping.

Figure 18.78 Kakula Backfill Plant Process Flow Schematic



Source: Patterson and Cook 2025

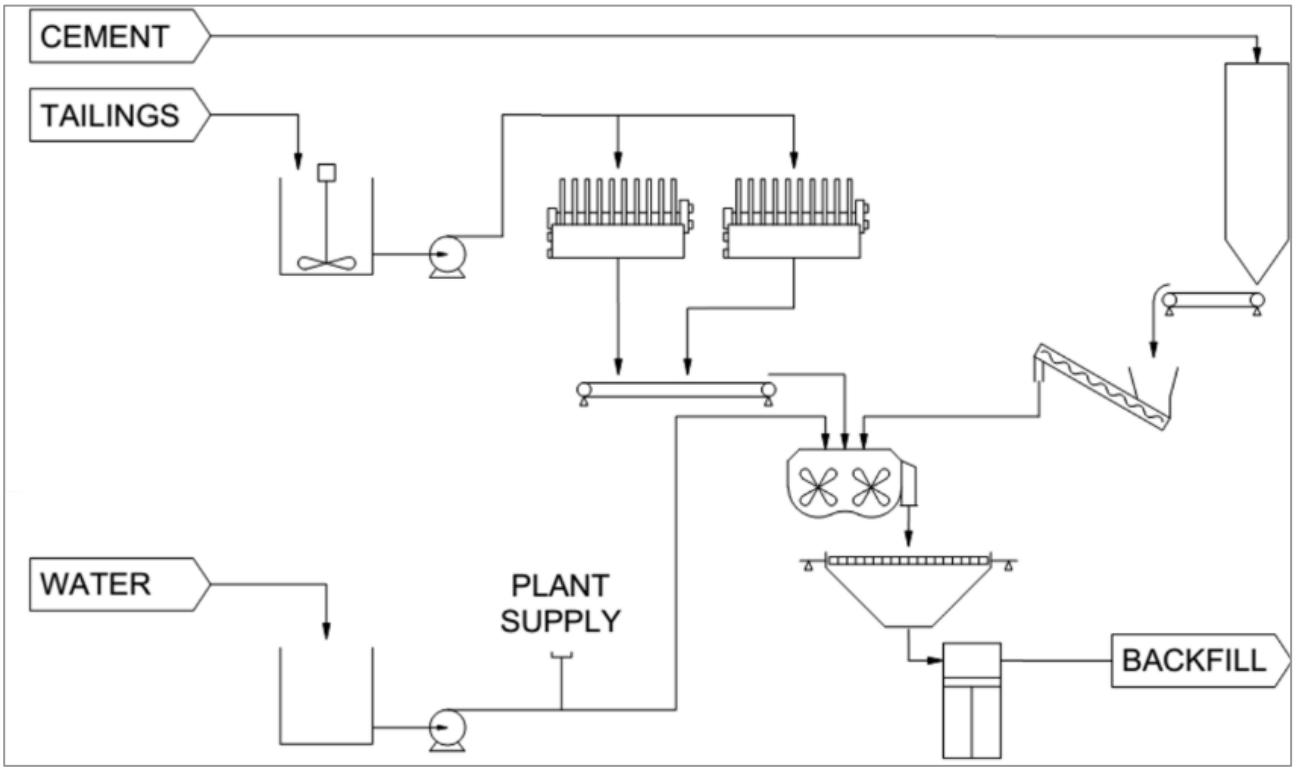
18.11.1.2 Kamoa 1 Backfill Plant

The Kamoa plant (Figure 18.79) is supplied with tailings from the Kamoa concentrator plant. Tailings are pumped from the tailings thickener to the backfill plant tailings storage tanks (one per module) through overland pipelines. The tailings tanks serve as a buffer to short-term tailings supply outages. Tailings are pumped to vacuum disc filters (two per module) for local dewatering. The disc filters discharge filter cake onto a filter cake conveyor which dumps into a continuous paste mixer. Trim slurry, water and binder are added to the mixer to create a homogenous CPB.

The binder storage system, consisting of one, 4000-tonne storage silo, is supplied directly from binder delivery trucks. Binder is transferred from the storage silo to either one of the two 750-tonne dosing silos (one per module) through pneumatic conveying. The dosing system consists of a weigh belt and a screw conveyor, which feeds binder to the continuous mixer.

The paste in the continuous mixer discharges by overflowing into a paste hopper. The hopper facilitates continuous flow of paste to the underground. Paste from the hopper reports to a 150-bar rated hydraulic piston pump, which distributes the paste through a DN 200 pastefill reticulation system to the mined stopes. The reticulation system consists of overland paste pipelines, surface to underground paste boreholes and underground reticulation piping.

Figure 18.79 Kamoa 1 Backfill Plant Process Flow Schematic

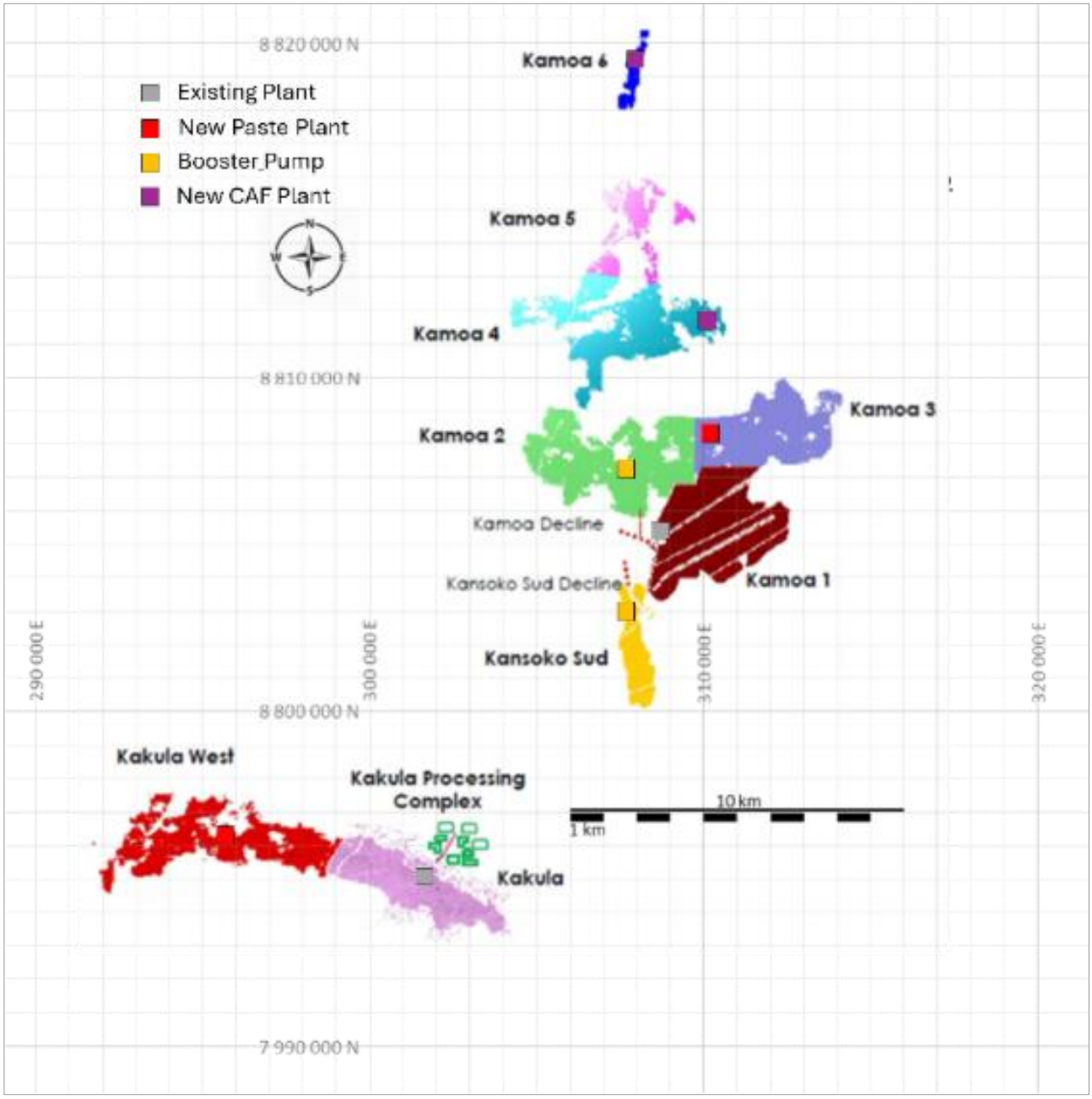


Source: Patterson and Cook 2025

18.11.2 Planned Backfill Systems

Development of additional orebodies within the Kamoa-Kakula complex is ongoing. As these orebodies come online, additional backfill capacity (Figure 18.80) will be required to meet future production demands.

Figure 18.80 Kamoa-Kakula Complex Backfill Plant Locations



Source: DRA 2023

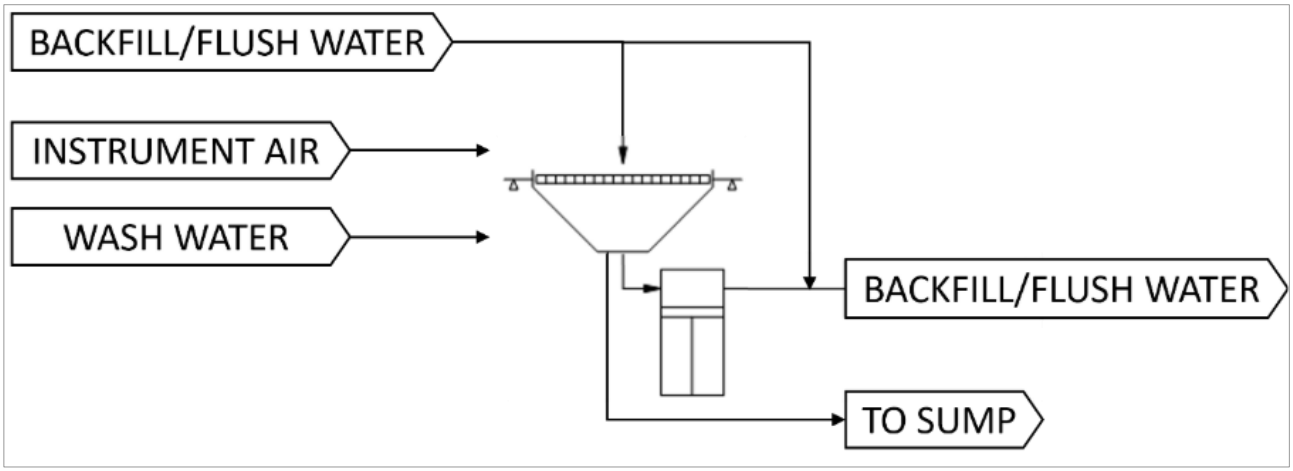
18.11.2.1 Planned Paste Booster Pump Stations

The Kansoko Sud and Kamoa 2 orebodies will be serviced by the existing Kamoa 1 Backfill Plant. To manage the longer overland pumping distances, paste booster pump stations (BPS) will be installed for Kansoko Sud in 2028 and for Kamoa 2 in 2031.

Produced paste is pumped from the backfill plant to the BPS (Figure 4) using a hydraulic piston pump. The BPS receives the paste in an agitated hopper, which provides buffering and maintains a continuous, consistent flow to the underground distribution system. From the hopper, a 150bar hydraulic piston pump

sends the paste into the DN 200 pastefill reticulation network for delivery to the mined stopes. The reticulation system consists of overland paste pipelines, surface to underground boreholes, and underground distribution piping.

Figure 18.81 Paste Booster Pump Station Process Flow Schematic



Source: Patterson and Cook 2025

18.11.2.2 Planned Cemented Backfill Plants

New backfill plants are planned for the Kamoa 3 and Kakula West orebodies. These systems will be implemented in phases and will follow the same design as the Kamoa 1 Backfill Plant. The Kamoa 3 plant will include three modules, with the first two commissioned in 2029 and the third added in 2031. The Kakula West plant will consist of two modules, with construction planned for 2032 and 2035.

18.11.2.3 Planned Cemented Aggregate Plants

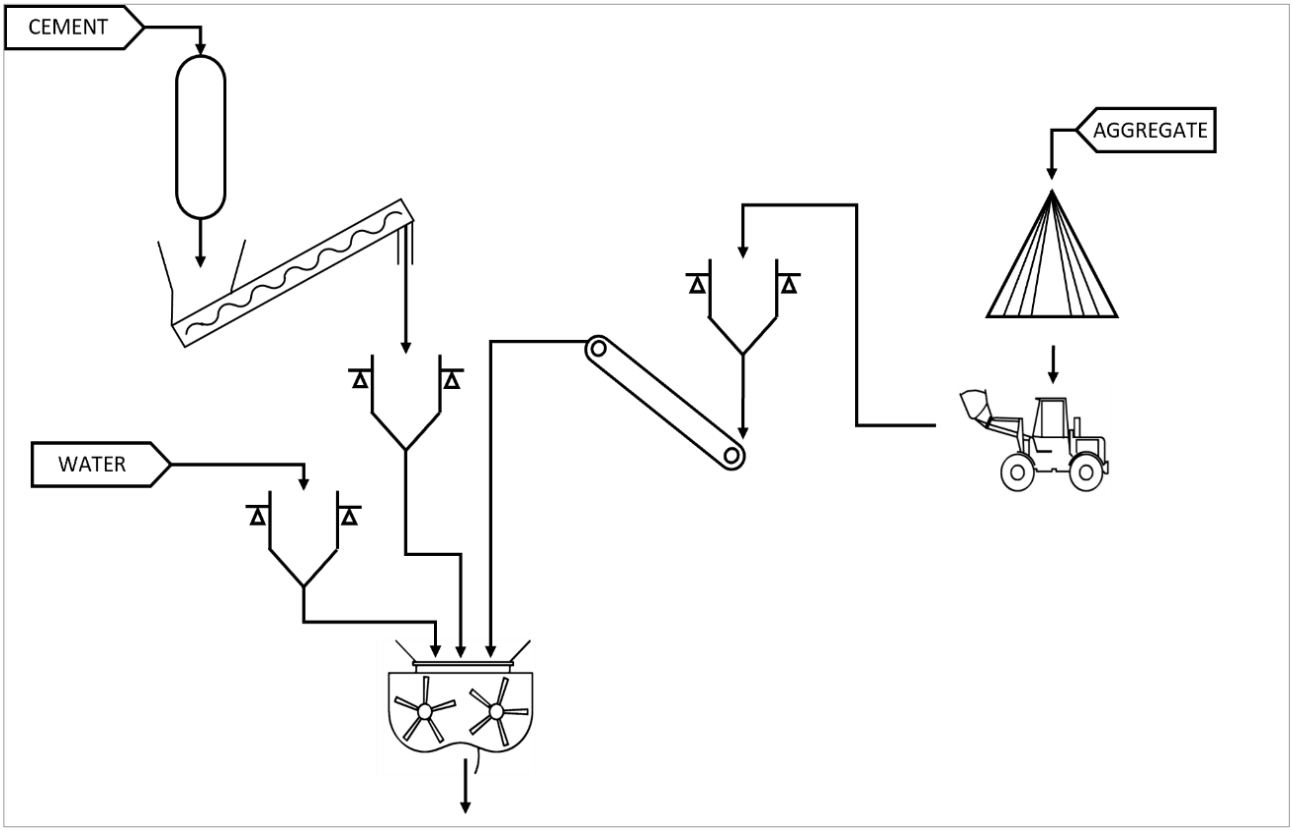
The satellite orebodies Kamoa 4, 5, and 6 require much reduced volumes of backfill compared to other deposits as these are mostly planned as R&P mining operations. The Bonanza portion of Kamoa 4 and a small portion of Kamoa 6 will be backfilled using Cemented Aggregate Fill (CAF) (see Figure 18.82). This approach is driven by the distance from the Kamoa Concentrator and lower overall backfill requirements. A single module batch mixing CAF plant, along with a mobile crushing unit, will be constructed for Kamoa 4 in 2029. The plant will later be relocated to the Kamoa 6 orebody in 2037. This relocation will shorten haul distances and improve efficiency toward the end of the LOM.

Waste rock is crushed using a mobile crushing plant to produce aggregate for Cemented Aggregate Fill (CAF). The aggregate is loaded into a feeding bin with a frontend loader. From the bin, the material is discharged onto a weigh belt, which feeds an incline conveyor. The conveyor transfers the aggregate into a batch mixer, where trim water and binder are added to produce the CAF mixture.

The binder system includes two 400tonne dosing silos, inclined screw conveyors, and a cement hopper. These components work together to supply binder to the batch mixer.

After each mixing cycle, the CAF is discharged from the mixer into a hopper. The hopper directs the material into an articulated dump truck (ADT). Once full, the ADT travels to the surface access point for the underground CAF borehole and dumps the material down the borehole. An underground ADT receives the CAF and transports it to the designated mined stopes for placement.

Figure 18.82 Cemented Aggregate Backfill Plant Process Flow Schematic



Source: Patterson and Cook 2025

19 Market studies and contracts

19.1 Market studies and offtake strategy

Kamoa-Kakula is transitioning its offtake strategy from the production and sale of high-grade copper concentrate to copper blister anodes, following the commissioning and ramp-up of a 500,000 tonne per annum direct-to-blister furnace (DBF) since late 2025.

In the past, Kamoa-Kakula produced and sold copper concentrate with a grade of approximately 50% Cu from the Kakula mine (Phase 1 and 2 concentrators) and approximately 35% Cu from the Kamoa mine (Phase 3 concentrator). Both concentrates have relatively low levels of deleterious elements and may be sold on internationally competitive concentrate terms.

Kamoa-Kakula has also produced blister copper via toll smelting at the Lualaba Copper Smelter (LCS), a custom smelting facility located approximately 60km from Kamoa-Kakula by road, with a five-year agreement in place for approximately 150,000 wet metric tonnes (WMT) of concentrate, which is transformed into blister copper and sold by Kamoa Copper SA (KCSA).

Since production commenced in 2021, KCSA has run tender processes for the sale of copper concentrate, blister copper and anodes, often taking into consideration concurrent pre-payment financing packages and well as other commercial considerations, including point of delivery and freight.

Initial agreements were signed in with CITIC Metal (HK) Limited (CITIC Metal) and Gold Mountains (H.K.) International Mining Company Limited (Gold Mountains), a subsidiary of Zijin Mining, for 50% each of the copper products produced by Phase 1, which was later expanded to include Phase 2.

A third offtaker, Trafigura Pte. Ltd. (Trafigura) was introduced in 2022 with a fixed volume agreement for copper concentrates.

In 2024, agreements were entered into with CITIC Metal and Gold Mountains for the sale of copper blister anodes (approximately 99.7% Cu), in advance of first production by the Kamoa-Kakula smelter, totaling 80% of anode production over a three-year term. In addition, in 2025 an agreement was signed with Trafigura for the remaining 20% of anode production.

KCSA's offtake agreements contain standard, international commercial terms, including payable metal values, as well as treatment charges (for the sale of copper concentrates only) and refining charges, both of which are determined by international benchmarks.

Offtake agreements have typically been on a free-carrier basis (FCA), meaning that the buyers are responsible for arranging freight and shipment from the mine gate to the final port of destination, which is reimbursed on an open-book basis.

KCSA's anodes, blister, and concentrates are exported via the ports of Durban in South Africa, Dar es Salaam in Tanzania, and Lobito in Angola, and to a lesser extent Walvis Bay in Namibia and Beira in Mozambique.

Kamoa Copper does not have any derivative contracts in place at Kamoa-Kakula.

Kamoa-Kakula's smelter is expected to produce up to 700,000 tonnes per annum of high-strength sulphuric acid as a by-product. Sulphuric acid is in high demand in the Central African Copperbelt where it

is consumed in the production process of copper and cobalt from oxide ores. Approximately 2 million tonnes of sulphur is imported into the Copperbelt region annually and transformed by sulphur burners into 6 million tonnes of acid, in addition to 2.5 million tonnes of acid produced directly by regional smelters such as Kamoa-Kakula. This provides an effective floor-price and captive end-market to sell the sulphuric acid to local mining operations through a mixture of short- and long-term contracts.

19.2 Copper market overview and dynamics

The biggest determinants of market prices can be broadly summarized 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.

Copper trades freely on global markets, including the London Metal Exchange (LME), Shanghai Futures Exchange, and the New York Commodity Exchange. Copper prices have long been seen as a reliable barometer of the global economy, and as such, are sensitive to global macro-economic developments resulting in cyclicality and volatility. Copper reached an all-time high on the LME of $6.13/lb in January 2026, driven by surging demand from data centers and supply disruptions, compared to the LME copper price low over the last ten years of $2.04/lb in June 2016.

Kamoa Copper has not conducted market studies for Kamoa-Kakula due to the producing nature of operations.

20 Environmental studies, permitting and social or community impact

20.1 Environmental baseline studies or environmental impact assessment (EIA)

KCSA has undertaken a comprehensive suite of environmental and social baseline studies and impact assessments. These studies started at the initial project development phase and have included subsequent staged expansions and additional supporting specific technical studies.

20.1.1 Summary of studies conducted

The Environmental and Social Impact Assessment (ESIA)updates continue to leverage existing baseline studies and impact assessments, integrating operational aspects and the most recent expansion impacts. It is subject to approval by the Directorate for the Protection of the Mining Environment (DPEM) (Kamoa 2025).

Baseline studies, monitoring management reports and other supporting information include:

- Baseline studies and impact assessments including air, water, soil, biodiversity, noise, health, and heritage aspects.
- Annual, quarterly, and monthly monitoring of all or some of the following items: water quality, radioactivity, biodiversity, air quality, and closure liability.
- Specialist studies for supporting infrastructure, such as power transmission and the Bulk Emulsion Plant.

These studies are derived from the Environmental and Social Impact Assessment (Revision A, April 2025, the ESIA) and its specialist annexes, which at times supersede prior ESIA and related references and other summaries). On other occasions the ESIA reuses legacy work where it is deemed relevant.

Where the ESIA cites legacy specialist work by WSP/Golder Associates (2017-2022), those same studies (and closely related ones) are already referenced in Section 20 of the IDP23 technical report as the input is comprehensive baseline studies and impact assessments. In other words, the ESIA carries forward with validated baselines and models, and only updates baseline and technical studies information where new work was undertaken, such as WSP 2024 model updates, Airshed/WKC 2024, and SRK 2024.

The ESIA integrates new specialist updates including WSP 2024 Tailings Storage Facility (TSF) geochemistry & numeric models, Airshed 2024 air emissions & dispersion, SRK 2024 ambient air, WKC 2024 noise and vibration, and KELKAM 2024 radiation. These studies, supersede or complement the previous ESIA's narrative where measurements and/or modelling have changed or been updated (Table 20.1).

Table 20.1 ESIA baseline and supporting technical studies update status

Component	Latest Source & ESIA Table/Figure	Status	Notes
Geology (relevant context)	OreWin (2023) – Tables 52-53; Figures 68-70	Carry-forward + update	Same as previous ESIA and embedded unchanged.
Soils & Land Use	Univ. Lubumbashi (2024) – Table 58; Figures 71-73	Carry-forward + update	Including WSP/Golder references (2021/22).
Climate and Air Quality	Airshed (2024) and CEMIC (2022) – Tables 59-60, 116; Figures 74-77	Carry-forward + update	Emissions inventory for smelter/generators.
Ambient Air Monitoring	SRK (2024) – Tables 61-66; Figures 78-86	Supersedes	2023-2024 monitoring replaces previous ESIA narrative and includes references to legislative requirements.

Component	Latest Source & ESIA Table/Figure	Status	Notes
Air Dispersion Modelling	Airshed (2024) – Tables 41-42; Figures 118-124	Supersedes	Updated smelter/generator scenarios and quantitative emissions information.
Noise and Vibration	WKC (2024) and CMS (2024) – Tables 67-78; Figures 87-90	Supersedes	New surveys, peak-particle velocity (PPV) and receptor compliance data complemented by continuous monitoring activity. Golder references (2022).
Surface Water	Kamoa (2022-2024), Golder (2017-2022), WSP, 2024 — Tables 79–82; Figs 91–94	Carry-forward + update	Consolidates 2022–2024 monitoring with continuous monitoring activity by Kamoa. Previous ESIA work referenced Golder.
Groundwater Quality	ESIA consolidated – Tables 83-84; Figures 95-99	Carry-forward + update	Kamoa continuous monitoring activity. Previous ESIA legacy Golder carried forward.
Hydrogeology (Conceptual)	Golder (2022), ESIA consolidated – Table 85; Figures 100-102	Carry-forward	Builds on Golder previous conceptual work.
Hydrogeology (Numerical)	WSP (2024) and Golder (2022) — Tables 90–94; Figs 103–107, 130–132	Carry-forward + update	New modelling. Previous ESIA references Golder.
TSF Tailings and Geochemical analysis	WSP (2024) and Epoch (2024-2023) — Tables 55–56; Figs 61–65; Figs 135–140	Supersedes	Adds dam breach analysis. Previous ESIA TSF context from Epoch.
Biodiversity (Fauna, Flora and Sensitive Environments)	CEMIC (2022) — Tables 95–103; Figs 108–114	Carry-forward	ESIA reuses 2022 baseline. Threatened species unchanged. Adds delineation of dambos and dilungu.
Radiation	KELKAM (2021-2024) and NECSA (2013) — Tables 6–13; Figs 13–20	Carry-forward + update	ESIA adds 2024 monitoring built over NECSA baseline.
Social Baseline	CEMIC (2022) and Golder (2020-2022) — Tables 104–112; Figs 115–117	Carry-forward + update	Adds 2024 consultations and builds on previous ESIA Golder work.
GHG Baseline	ESIA Title IV; Hatch/Golder legacy work	Carry-forward	Reflects new power matrix and smelter impacts.

Source: ESIA, Knight-Piésold, 2025.

20.1.2 EIA status

Previously, Kamoa had completed an update of the ESIA and Environmental and Social Management Plan (ESMP) by commissioning Congo Environment and Mining Consulting (CEMIC) SARL for its Phase 3 expansion in 2022. CEMIC conducted own data collection and analysis and included information provided by WSP (formerly Golder) on past environmental and social baseline and impact assessment studies. Recently, Kamoa commissioned Knight Piésold Consulting RDC SARL in April 2025 to complete the now current ESIA and ESMP. The ESIA is final and has been issued on April 2025 to the DPEM.

In line with DRC Mining Regulations and international standards, the 2025 update is compliant with current Mining Regulations' Article 463 of Decree No. 038/2003, as amended by Decree 18/024 which requires companies to revise impact assessments and management plans periodically or whenever material changes occur.

The ESIA covers all operational and proposed expansions, supported by a comprehensive suite of baseline studies for environmental and social conditions (20.1.1 Summary of studies conducted). The social impact assessment was based on documented stakeholder consultation. A mine closure and rehabilitation plan

along with financial provisions are included. Monthly environmental monitoring reports provide ongoing data on air, water, noise, and biodiversity, supporting compliance and adaptive management.

20.1.3 Key findings

The overall outcome reported by Knight Piésold in the ESIA does not raise any potential fatal flaw concerns or impacts that cannot be mitigated to acceptable levels that could materially impact Kamoa's ability to extract the Mineral Resources and Mineral Reserves.

The ESIA identifies a range of environmental and social impacts, both positive and negative, across construction, operation, and decommissioning, including:

- Land access, permitting, and relocation
- Water management and hydrogeology
- Acid Rock Drainage (ARD) and tailings management
- Biodiversity, sensitive receptors, and protected areas
- Air quality, noise, and vibration
- Closure, legacy issues, and financial provisioning
- Social license, stakeholder engagement, and community health

Furthermore, several prior radiation assessments have been documented and reported. These assessments (NECSA 2013, Kelkam 2019, 2021, 2022) confirm normal radiation conditions and limited radiological risk. Annual monitoring is in place, and mining activities pose no enhanced background radiological risk.

20.1.3.1 Land access, permitting, and relocation

Any potential delays or disputes in land access, resettlement, or permitting could prevent or postpone mining in key areas, directly impacting resource/reserve extraction schedules.

Land tenure and community relocation

Soils are generally severely limited for agriculture, with main uses being subsistence cropping and charcoal production. Fundamentally, land use is influenced by topography, water access, and proximity to settlements.

The ESIA confirms that all mining and infrastructure areas are within the Kamoa Copper SA concession, with no third-party mining rights. However, ongoing and future expansions (notably any potential open pit, solar farm, and TSF expansions) require physical and economic resettlement of local communities.

The 2025 update has reduced the scale of planned resettlement (from 1,300 to 12 households for Phase 3) with renewed considerations of underground mining as opposed to surface mining resulting from the Q4 2024 demographic, socio-economic and property surveys.

Any delays or disputes in land acquisition, compensation, or resettlement could directly constrain access to ore zones and infrastructure corridors.

Permitting

The ESIA and ESMP are compliant with DRC Mining Regulations and with a focus to meet international standards (IFC). All planned expansions are subject to DPEM and Congolese Environment Agency (ACE) approval.

Any failure to secure timely environmental approvals could delay or restrict mining activities.

20.1.3.2 Water management and hydrogeology

Any potential inadequate dewatering, water management, or related regulatory non-compliance could result in flooding, environmental penalties, or suspension of mining activities.

Dewatering and groundwater drawdown

The ESIA's updated hydrogeological model (WSP 2024) confirms that extensive dewatering from both underground and open-pit operations can cause drawdown cones to potentially affect local water tables, with possible impacts on community wells and surface water baseflow.

The model predicts that, with mitigation, impacts are manageable. Any failure of dewatering systems or unforeseen hydrogeological conditions could affect safe work conditions potentially halting or restricting mining.

Surface water and flooding

The project area is subject to intense seasonal rainfall. The ESIA identifies the need for robust stormwater management, river and stream diversions including the Mbulamema stream to be diverted away from the proposed open pit mine. Any proposed open pit requires lined water control structures particularly in consideration of any material which is identified as acid generating.

Inadequate surface water management could result in pit flooding, TSF overtopping, watershed contamination, or infrastructure damage, leading to operational shutdowns.

Water quality and discharge

Effluent and mine water discharges in general comply with DRC standards for quality. Occasional exceedances in TSS (total suspended solids) are normally detected during rainy season. Contaminants (oil and grease) are generally attributed community sources.

Naturally occurring low pH levels are typically observed at some reference points, which, along with high TTS, can be associated with geological influences.

Any non-compliance from mining could result in regulatory action or suspension of discharge permits.

20.1.3.3 Acid rock drainage (ARD) and tailings management

ARD discharges or TSF failures could result in environmental contamination, loss of social license, and regulatory suspension of operations.

ARD potential

The ESIA confirms that Kamoa Pyritic Siltstone (KPS) waste as well as ore stockpiles, ROM and KPS Stockpile, and upper diamictite waste that may generate ARD (generated from the Kamoa 1 and 2 mine)

must all be treated as potentially acid generating. The open pit stormwater run-off areas, waste rock dump and ore stockpiles pads must be managed with lined facilities, water collection, and treatment plants (HDS process).

Failure to contain and treat ARD could result in groundwater and surface water contamination, regulatory penalties, or forced cessation of activities in affected areas.

Tailings storage facilities (TSF)

The Kakula TSF (Cells 1–3) and future Mupenda TSF are designed with partial lining and engineered stormwater controls. The ESIA's updated contaminant plume modelling shows that, with current mitigation, contaminant migration is limited.

The planned Hydromining of historic tailings (Kakula TSF Cell 1) for reprocessing follows a trend of sustainable mining practices by enabling further resources utilization and potentially supporting the management of the TSF related environmental impacts. Hydromining will draw supernatant water from Kakula TSF Cells 2 and/or 3 introducing additional water management and associated ARD risks.

Critical TSF breach or liner failure could still result in catastrophic environmental damage, loss of storage capacity, and regulatory shutdowns, despite mitigation measures.

20.1.3.4 Biodiversity, sensitive receptors, and protected areas

Disturbance of sensitive environments or heritage sites could result in regulatory or community-imposed work stoppages, additional mitigation costs, or loss of access to key mining areas.

Sensitive environments

While no officially protected areas are within the concession, the ESIA identifies dambos, dilungu, and certain wetlands as ecologically sensitive.

Partially protected fauna such as the Crocodylus niloticus and Python sebae are present. Expansion into these areas requires strict mitigation of potential critical risks and ongoing monitoring. The conclusion from the faunal surveys completed indicate such depauperate assemblages are the result of historical hunting and habitat transformation.

Rare and endangered flora, such as Protea micans, Protea angolensis, Vanilla unifolia, Xerophyta equisitoides, and Encephalartos schaijesii, representing the uniqueness of their original habitat while providing ecological value and biodiversity. Over 230 plant species have been recorded; three of which are classified IUCN Red Data species. Critical habitat is defined by the presence of orchid Habenaria hebes, found only on dilungu (unique wetland ecosystem).

The spread of invasive plants and animals can increase in disturbed areas due to Kamoa-Kakula's ongoing activities and planned expansion.

Unintended consequences of failing to control invasive species or protect sensitive environments could trigger regulatory intervention or additional area offsets requirements.

Heritage and sacred sites

Over 70 cemeteries, 21 archaeological sites, and 25 sacred sites are mapped from extensive surveying completed to date.

Any unplanned disturbance could result in work stoppages, legal action, or reputational harm.

20.1.3.5 Air quality, noise, and vibration

Exceedances in air, noise, or vibration limits could result in regulatory penalties, community opposition, or operational restrictions.

Atmospheric emissions

Kamoa is inherently exposed to several emissions sources (e.g., smelter/refinery, sulphuric acid plant, oxygen plant, concentrator, waste storage, vehicles, explosives, back-up energy generators, and dust).

The ESIA confirms that, under normal operations, air emissions (SO_2, NO_2, PM_{10}, H_2SO_4) are within DRC and IFC standards at sensitive receptors. However, due to the nature of the activities, exceedances may occur during abnormal events. Non-compliance could result in fines or operational restrictions.

Noise and vibration

Blasting and heavy equipment generate noise and vibration but monitoring shows compliance with DRC limits. However, planned expansion constructions activities are expected to increase blasting near communities which in turn could trigger complaints and/or eventual regulatory action.

20.1.3.6 Closure, legacy issues, and financial provisioning

Insufficient closure planning or legacy contamination could result in loss of mining rights, additional bonding requirements, or reputational harm.

Closure planning

The ESIA provides a detailed closure plan and outcome financial provision of $330 M for closure and rehabilitation costs by the end of the mine life and for 10-years beyond cessation of production activities, covering rehabilitation, water treatment, and post-closure monitoring.

Any underestimation of closure liabilities or failure to execute closure plans could result in regulatory non-compliance or loss of mining rights.

Legacy contamination

No new legacy liabilities have been identified, but ongoing ARD, tailings, or hydrocarbon contamination could create future liabilities if not managed.

20.1.3.7 Social license, stakeholder engagement, and community health

Loss of social license or community health crises could result in operational disruptions or additional mitigation costs.

Stakeholder engagement

The area is characterized by rural villages, with livelihoods based on subsistence farming and charcoal production. Formal employment is low making Kamoa a significant employer. Artisanal mining is not present within the Kamoa footprint; however, it is evident in the region.

The ESIA documents extensive consultation, with relocation, water access, and ecosystem services as dominant concerns.

Failure to maintain social license could result in protests, blockades, or reputational damage.

Community health

The influx of workers and population growth increases pressure on health, water, and sanitation infrastructure.

Potential outbreaks of disease or perceived health impacts could trigger regulatory or community action.

20.1.4 Mitigation measures proposed

The ESMP details mitigation and enhancement strategies for all significant impacts, including dust and noise controls, water management, biodiversity protection, and community health and safety measures. These include, but are not limited to:

- Implementation of an Environmental and Social Management System (ESMS) using Isometrix software for monitoring and reporting.
- Ongoing stakeholder engagement and grievance mechanisms.
- Sustainable development plan (SDP) with a five-year social project budget of approximately US$8.6M, including education, health, infrastructure, and livelihoods projects.
- Water management strategies, stormwater management, and effluent treatment to meet DRC standards.
- Biodiversity management, including protection of critical habitats and ongoing botanical surveys.
- Resettlement and rehabilitation policy framework guided by national legislation and IFC standards, with multiple phases of resettlement completed for affected households.

Monitoring parameters are aligned with DRC legal requirements as demonstrated by ongoing environmental reports.

20.2 Waste, tailings, water and monitoring plans

Kamoa-Kakula implements a comprehensive waste and tailings management strategy designed to minimize environmental risks and ensuring regulatory compliance throughout the mine life cycle. Waste rock is managed through a combination of materials consumption for construction use, lined storage for potentially acid-generating (PAG) material, and progressive rehabilitation. Tailings are managed in engineered tailings storage facilities (TSF), with the current Kakula TSF comprising three cells and a future Mupenda TSF planned for additional capacity. Both facilities incorporate robust design features such as partial HDPE lining, downstream embankment construction, and stormwater diversion to mitigate seepage and protect groundwater and surface water resources.

20.2.1 Waste and tailings disposal

Tailings storage facilities

Kakula TSF (Cells 1-3) and Mupenda TSF comprise the current and planned tailings storage facilities system; Kakula Cell 1 is expected to reach capacity in 2028, Cell 2 will be commissioned in 2026 followed by Cell 3 a year later and Mupenda, East of Kamoa which is expected to begin operations in 2029.

The tailings produced are classified as non-acid generating and not radioactive, with leachability for copper and iron managed through ongoing geochemical testing and water quality monitoring. The project also plans for the co-disposal of smelter waste and the hydro-mining of historic tailings, further supporting resource recovery and environmental stewardship. Closure and post-closure strategies are integrated into facility design, with financial provisions established to ensure long-term stability and compliance.

Waste rock dumps (WRD)

Waste rock dumps present a lower environmental risk, as the majority of waste rock is repurposed for on-site construction. New WRDs are planned as a function of future expansion with the proposed open pit at Kamoa 2 having a new waste rock dumps developed to handle material from the shallow oxide, transitional, and sulphide ore extraction. The objective is to maintain ecological balance. environments. All KPS containing WRD will be lined with a clay pad.

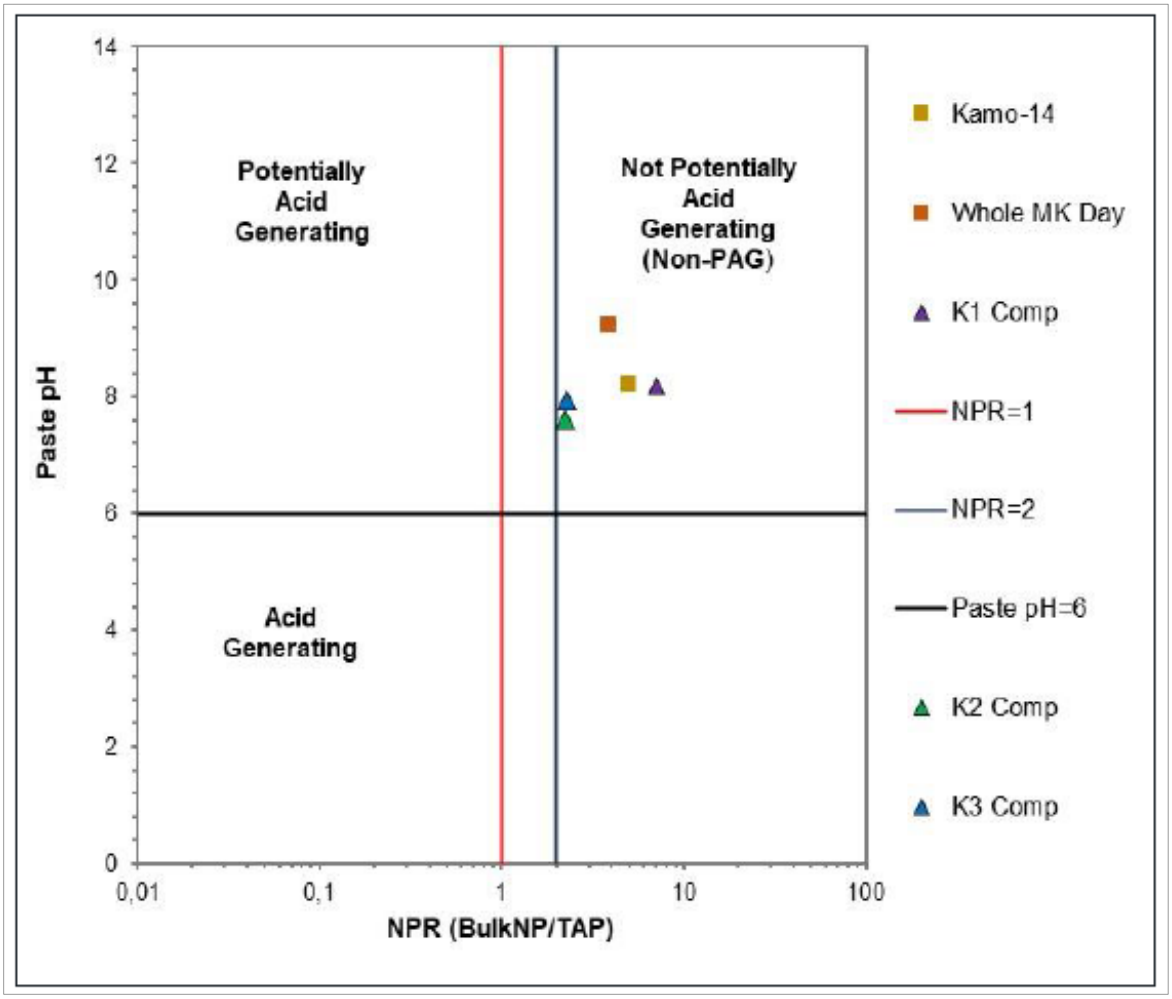
Geochemical characterization

The ESIA concludes from the geochemical assessment conducted by WSP (2024) that Phase 3 tailings are non-potentially acid generating and non-radioactive, meeting safety standards (Figure 20.1). While copper and iron concentrations in the tailings exceed low-risk thresholds – indicating some potential for metal leachability – all other chemical components are within acceptable limits, with minimal organic leaching observed. A review of ultra-fine copper tailings from Project 95[1] revealed a particle size distribution dominated by fine particles and copper oxides/carbonates, resulting in lower sulphide content and reduced acid generation potential compared to current Kakula tailings.

Kamoa plans to co-dispose copper slag waste with tailings slurry and apply dry deposition of gypsum and neutralized sludge at the TSF, with a total smelter waste volume of around 0.6 Mtpa. These waste streams were assessed – slag, gypsum, and neutralized sludge – as likely leachable for metals such as Cd, Cu, Pb, As, and Zn, based on preliminary modelling and literature data. The integration of slag into the tailings slurry could affect leaching behavior, and further characterization of site-specific smelter waste is recommended to refine geochemical predictions and validate classifications.

[1] "Addition of new equipment and rerouting of some product streams to improve copper recovery to 95% at Kakula." (ESIA Knight Piésold, 2025).

Figure 20.1 Classification of Kamoa and Kakula tailings



Source: WSP, 2024

20.2.2 Water management

Water management at Kamoa-Kakula is structured around minimizing freshwater consumption, maximizing water recycling, and protecting both surface and groundwater resources. The TSFs are designed to return supernatant water to the processing plants, with an average return rate of approximately 8,748 m^3/day (about 89% of water sent to the TSF). Stormwater is managed through a network of diversion trenches and settling ponds, ensuring separation of clean and contact water and compliance with DRC regulations for effluent discharge.

Acid water generated from mining activities, particularly from KPS waste and ore stockpiles, is collected in lined ponds and treated using high-density sludge (HDS) processes before discharge. The water balance is regularly updated to account for mine expansion, rainfall variability, and operational changes. Predictive groundwater and contaminant transport models inform the design and operation of water management infrastructure, including the sizing of pumps, sumps, and treatment plants.

20.2.3 Site monitoring programs

A comprehensive environmental monitoring program is implemented to ensure ongoing compliance and early detection of potential impacts. Water quality (surface and groundwater) is the key parameter monitored.

Groundwater and surface water are sampled at multiple locations upstream and downstream of the TSFs and waste facilities, with results compared to DRC and World Health Organization standards (WHO, 2017 guidelines). Piezometers and monitoring boreholes are installed around TSFs to track phreatic levels and detect any potential seepage.

Air quality is monitored for particulate matter (inhalable particulate matter PM_{10}, $PM_{2.5}$), SO_2, NO_2, and metals, with results generally within regulatory thresholds.

20.2.4 Post-closure monitoring

Post-closure monitoring program will be implemented for a minimum of 10 years following closure, or as required by regulatory authorities. The program will focus on the following key indicators:

- Physical Stability: Regular inspections of embankments, slopes, and drainage structures to detect signs of erosion, settlement, or instability. Piezometers and settlement markers will be monitored to confirm embankment integrity.
- Surface Water Quality: Routine sampling of runoff and discharge points downstream of the TSF to ensure compliance with DRC and international water quality standards. Monitoring will focus on parameters such as pH, total suspended solids (TSS), metals (Cu, Fe, Zn, Pb, As), and hydrocarbons, as per broader surface water quality monitoring program.
- Groundwater Quality: Monitoring boreholes upstream and downstream of the TSF will be sampled quarterly for key indicators, including pH, TDS, sulphate, nitrate, and metals. The extent and concentration of any contaminant plumes will be tracked, with particular attention to areas downgradient of the TSF and in proximity to community water sources, as per broader groundwater quality monitoring program.
- Vegetation and Erosion: Biannual visual inspections and photographic records will be used to assess the success of revegetation and the effectiveness of erosion control measures. Areas of poor vegetation establishment or active erosion will be targeted for remedial action, as per broader rehabilitated areas monitoring program.
- Seepage and Leachate: Seepage collection systems will be monitored for flow rates and water quality. Any increase in seepage or deterioration in water quality will trigger investigation and, if necessary, additional mitigation.

20.3 Permitting requirements

The Kamoa-Kakula Project is subject to a comprehensive permitting regime that governs all aspects of exploration, construction, operation, and closure. The permitting process is anchored in the DRC Mining Code and Mining Regulations, which require periodic updates to the ESIA and ESMP to reflect project expansions and regulatory changes.

All major project infrastructure – including mining, processing, waste, and tailings facilities – are covered by valid environmental approvals and are subject to ongoing oversight by the DPEM and the ACE.

The ESIA confirms that Kamoa maintains compliance with applicable national and international standards, including the IFC Performance Standards. The permitting process is supported by detailed baseline studies, stakeholder engagement, and the submission of technical documentation for each project phase. Compliance is monitored through regular reporting, site inspections, and adaptive management, ensuring that the project remains in good standing with regulatory authorities and meets its environmental and social obligations.

20.3.1 Regulatory framework

The project is governed by the DRC Mining Code and associated regulations, with oversight by DPEM and other authorities according to the following framework (Table 20.2):

- DRC Mining Code (Law No. 007/2002) and Mining Regulations (Decree No. 038/2003, as amended by Decree 18/024): These set out the requirements for environmental and social impact assessment, management planning, and periodic revision of permits and approvals. Article 463 of the Mining Regulations specifically requires that the ESIA and ESMP be updated whenever material changes to mining or processing activities occur.
- Environmental and Social Impact Assessment (ESIA) and Environmental and Social Management Plan (ESMP): These documents must be approved by the DPEM and ACE prior to the implementation of any new or expanded facilities.
- International Standards: Additionally, Kamoa seeks to align its permitting and management practices with the IFC Performance Standards, Equator Principles, and World Bank EHS Guidelines, particularly for stakeholder engagement, environmental monitoring, and closure planning.

Table 20.2 Summary regulatory framework

Regulatory Instrument	Description	Application
DRC Mining Code	Primary mining law	All mining, waste, TSF
Mining Regulations	Technical standards, permitting	ESIA/ESMP, monitoring, closure
IFC/Equator Principles	International standards	ESIA/ESMP, stakeholder engagement, closure

Source: ESIA Knight-Piésold (2025), Kamoa (2025).

All exploration, mining, and quarrying activities are required to have an approved environmental plan in place, ensuring responsible management of environmental impacts throughout the project lifecycle. Operators are obligated to adhere to the rehabilitation requirements specified in the plan, demonstrating their commitment to restoring sites following operational activities.

To guarantee that rehabilitation obligations are met, applicants must provide a financial guarantee, typically in the form of a bank guarantee. These secured funds are reserved exclusively for site rehabilitation and remain inaccessible to the DPEM, underscoring their intended use.

Kamoa has fulfilled its rehabilitation guarantee requirements in accordance with instalments detailed in the project's updated environmental impact study.

20.3.2 List of required permits

The following permits and approvals listed are required (Table 20.3) and are maintained by the Kamoa's Legal department and tracked on its Permit Register (Kamoa, 2025):

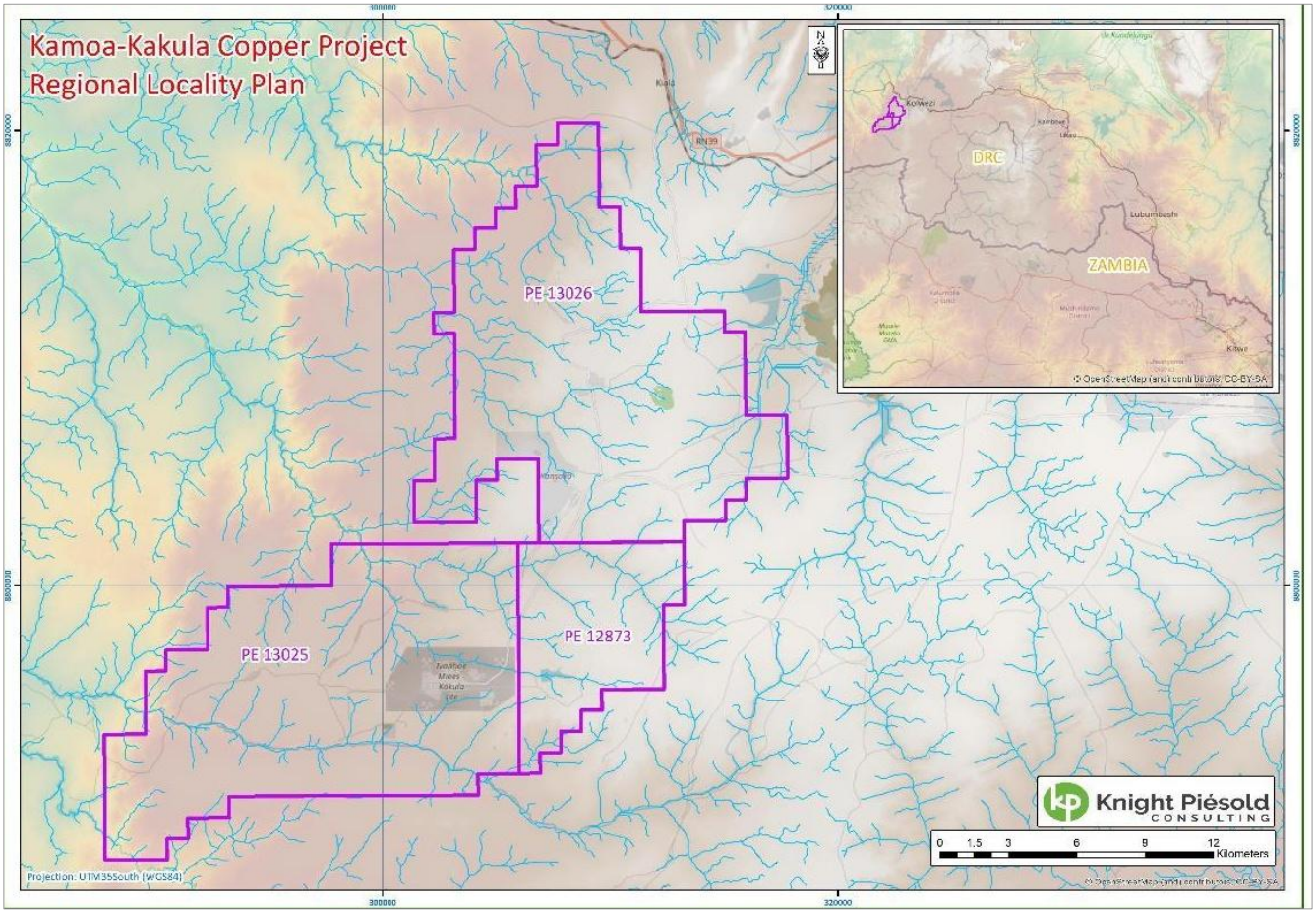
- Exploitation Permits (PE): Covering all mining areas, valid through 2042 (Figure 20.1).
- Environmental Permits: Associated to the approval of the ESIA/ESMP for all major project phases and expansions.
- Financial Guarantee for Rehabilitation: Provides a financial guarantee to cover rehabilitation costs.
- Annual Payment of Surface Rights and Taxes: Required to maintain the validity of the exploitation permit.
- Classified Facilities, Explosives Storage and Handling as well as Hazardous Waste: For facilities presenting environmental or safety risks.
- Social Obligations: Cahier des Charges (requirements document) of social projects responsibilities and associated public consultation.
- Various other supporting infrastructure, hydrocarbons, and telecommunications permits.

Table 20.3 Kamoa permit register – summary of key permits.

Permit	Authority	Status	Expiry	Notes
Exploitation Permits (PE) PE 12873, PE 13025, PE 13026	Ministry of Mines	OBTAINED	19 August 2042	Renewal 5 years (earliest) to 1 year (latest) before expiry.
Feasibility Study	Mining Register (CAMI) Directorate General of Taxation (DGI)	OBTAINED Renewal underway	TBD	FS 2023 submitted April 2023, approved October 2023. Update due 2025.
Permit to Sell Marketable Mining Products	Ministry of Mines	OBTAINED	31 March 2026	Subject to copper grade requirements. 18 October 2025 exceptional authorization to export concentrate (valid for 1 year).
Annual Surface Rights	Mining Register (CAMI)	PAID FOR	31 March 2026	Paid yearly to CAMI by 31 January.
Certificate No. 0165/ACE/DG/LO/AIE/MK/2024	DPEM/ACE	OBTAINED	13 July 2027	Current issued 20 September 2024. Updated ESIA subject to approval by the DPEM.
Financial Guarantees	DPEM	PAID FOR	31 December 2025	Annual payment within 30 days from EISA/ESMP anniversary.
Cahier des Charges	Provincial Governor / Ministry of Mines	SUBMITTED	31 December 2025	Approval request drafted and submitted. Program execution follow-up with the Territory Administrator/Mayor is in preparation.

Source: ESIA Knight-Piésold (2025); Kamoa Monthly Environmental Reports; Kamoa Permit Register (English_ KAMCO PERMIT REGISTER_ML_DB, 2025.xlsx).

Figure 20.2 Kamoa mining licenses



20.3.3 Conditions and other obligations

All project facilities are located within the Kamoa Copper SA concession and there are no other third-party mining rights within the permitted area or the immediate vicinity. Surface rights and land access are documented and maintained through a formal permit register (Kamoa, 2025).

Facilities must be constructed to engineered standards, with appropriate lining, seepage collection, and stormwater diversion, in accordance with DRC regulations and international best practice.

Regular monitoring of water quality, embankment stability, and seepage is required, in alignment with Annex VIII of the Mining Regulations and with results reported to DPEM and ACE accordingly. Any exceedances must be addressed.

Kamoa is required to maintain a financial guarantee for closure and rehabilitation, with the current provision for TSF aspects at $15.45 million.

Ongoing consultation with affected communities, including grievance mechanisms and public disclosure of monitoring results. Economic compensation for agriculture fields is being undertaken according to Annex XVIII of the Regulations in terms of the Mining Code.

The ESMP must be updated as needed to reflect operational changes, monitoring results, and regulatory updates. Corrective actions identified for each expected potential impact must be implemented in response to non-compliance or emerging risks. The ESIA must align with various expectations set in Annex VIII of the Mining Regulations and contain an ESMP within, in alignment with Article 452 of the regulations. Additionally, Article 463 requires revisions to the ESIA/ESMP when change in activities warrant an amendment to the previously assessed environmental impacts (identified and analyzed as per Article 39 to 42 of Annex VIII of the Mining Regulations).

The ESIA details proposed mitigating controls to be put in place for each of the environmental impacts identified (Section “Title V: Mitigation and Rehabilitation Measures Programme”) as per expectations set in various Articles of Annex VIII of the Mining Regulations.

20.3.4 Risks and critical path dependencies

The DRC regulatory framework involves several critical path dependencies and risks that could disrupt operations and advancement of new project development and execution at Kamoa. The most significant dependencies include the timely renewal of key permits (such as the Mining Licences, Environmental Certificate, and other critical supporting permits), the submission and approval of updated ESIAs/ESMPs following each major change to the operations, and the completion of stakeholder consultations as required by DRC law and international standards.

Traditional permitting related potential risks include:

- Delays in permit renewals or approvals: Any delay in the submission or approval of updated ESIA/ESMP, or in the renewal of mining or environmental permits, could result in project delays or interruptions to operations.
- Changes in regulatory requirements: Amendments to the DRC Mining Code or environmental regulations may necessitate additional studies, public consultations, or modifications to project design and management plans.
- Stakeholder or community opposition: Failure to adequately address stakeholder concerns during the permitting process, particularly regarding land access, resettlement, or environmental impacts, could result in appeals, legal challenges, or social unrest, potentially delaying permit issuance.
- Interdependencies with expansion projects: The approval of new infrastructure (e.g., Mupenda TSF, Kamoa 2 Open Pit, solar farm) is contingent on the successful completion and approval of updated ESIA/ESMP and associated permits.

Kamoa is proactive in submitting applications, making payments, and following up with permitting related actions, however, faces delays and legal ambiguities in some areas.

- Annual payments for mining rights, surface rights, and area taxes are up to date, but issues with some documentation and registers regarding annual surface rights remain.
- Environmental and social compliance is generally up to date and the latest ESIA/ESMP update is in progress (submitted April 2025).
- The Cahier des Charges amendment is pending approval (due December 2025).
- Explosives and hazardous materials permits and certificates are actively managed, but some are pending or require renewal.
- Infrastructure and utilities permits (e.g., surface infrastructure, hydrocarbons storage, and electricity) are expired, lacking, or pending, with legal advice being sought for unclear requirements.

- Telecommunications and unmanned aerial vehicles (UAV) compliance permit applications and payments are made, but some certificates are missing or pending.

20.3.5 Compliance history

Kamoa has maintained satisfactory compliance record with respect to permitting and environmental obligations. Its environmental and mining permits have been consistently renewed and updated in line with regulatory requirements. The latest ESIA and supporting documentation confirm that all major permits are current and that there have been no significant violations or enforcement actions reported by the DPEM or ACE.

The ESIA/ESMP sets clear expectations regarding routine inspections, audits, and environmental monitoring reports that must be submitted as required with any minor non-conformances identified promptly addressed through corrective actions.

Kamoa's compliance history is further supported by the absence of major incidents or regulatory penalties related to environmental, social, or permitting matters. No significant violations reported; all major permits are current.

20.3.6 Financial assurance

Financial assurance for environmental rehabilitation and closure is a regulatory requirement under the DRC Mining Code and is a condition of the project's environmental approvals. The latest ESIA (Kamoa, 2025) provides a detailed closure cost estimate, prepared in accordance with the South African Department of Mineral Resources (DMR) guideline, and includes all direct and indirect costs associated with site rehabilitation, infrastructure removal, water management, post-closure monitoring, and contingencies.

The financial guarantee is provided through annual allocations that are updated periodically to reflect changes in site conditions, project scope, and regulatory requirements.

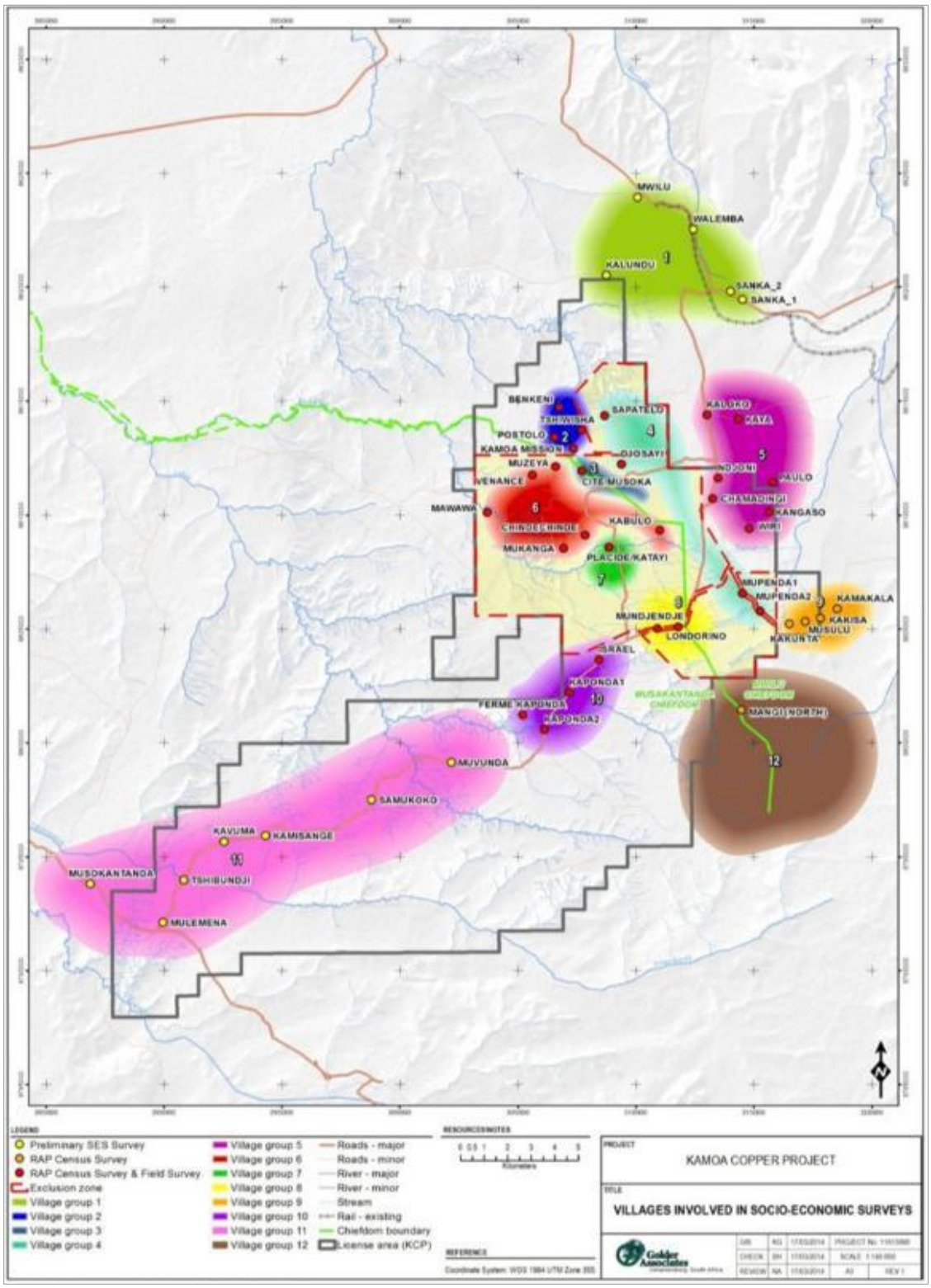
20.4 Social and community impact

The social and community impact assessment is a critical component in ensuring adequate responsible closure practices and balanced long-term benefits for affected stakeholders. Kamoa's diverse social area of influence highlights the business significant impact on local well-being, livelihoods, and development opportunities. The ESIA work has completed community engagement essential in the identification and addressing of challenges associated with mine closure, relocation, and ongoing site management.

20.4.1 Stakeholder identification

The Kamoa-Kakula Project's social area of influence encompasses approximately 41 communities (about 21,000 people), under two chiefs and two land leaders, within the Mutshatsha Territory (Luilu and Lufupa sectors) (Figure 20.3).

Figure 20.3 Villages involved in surveys



Source: Golder Associates

Stakeholders include representatives of communities and individual leaders and representatives from non-governmental (NGOs) and non-commercial organizations.

20.4.2 Engagement activities

Extensive stakeholder engagement was conducted in October 2024 as part of the development of the latest ESIA over meetings with traditional leaders, NGOs, and community members.

Key issues raised included relocation, water access, health, employment, and infrastructure. The engagement process followed DRC legal requirements (Act N°11/009, Mining Regulations Article 541/542) and included targeted meetings, information dissemination in French and Swahili, and a formal grievance mechanism.

Issues raised were naturally influenced by relocation matters as Kamoa had been completing relocation activities at the time of the survey. Other issues included water, health, jobs, education, infrastructure and land use.

20.4.3 Community agreements

Ongoing relocation and compensation programs are in place, with benefits including jobs, schools, clinics, and infrastructure.

Resettlement Action Plans (RAPs) have been implemented for affected households, in compliance with Mining Code Article 281, and Mining Regulations Annex XVIII of the DRC mining legislation. Funds committed through the various RAPs amount to $9,154,666 including a more recent 3-year program covering agriculture, livestock, fish farming, and financial management training budgeted at $2,826,800.

The recent demographic and property surveys led to a revised relocation plan, reducing the number of affected households from 1,300 to 12 for Phase 3, due to a shift from open pit to underground mining that resulted in only 900 hectares being impacted, as opposed to the previously estimated 9,300 hectares.

Community development programs include long term focused programs supported by the Cahier des Charges, which is a 5-year community development agreement part of the conditions to obtain and maintain mining rights. The current plan for Kamoa was approved in 2021 with a total budget $8,571,453 and covering the period from 2021 to 2025.

20.4.4 Local procurement and employment plans

Kamoa has established local hiring, training, and entrepreneurship programs, supporting agriculture, livestock, and small businesses. The company is a significant employer in the region, with formal employment opportunities prioritized for local residents.

As part of the Cahier des Charges, the establishment of the Centre of Excellence provides access to tertiary education opportunities and supports local procurement targets are set in line with the.

20.4.5 Cultural heritage management

According to the ESIA, comprehensive assessments across the concession have identified over 70 cemeteries, 21 archaeological sites, and 25 sacred sites within the mine's concession area. Cultural heritage baseline studies were completed in 2010, 2012, 2013, 2016, 2019, 2021, and 2022 by different ESIA updates or focused studies with conclusion pointing to low archaeological potential.

Protocols for discovery and protection are in place, including mapping, fencing, and community consultation. Sacred sites include waterfalls, ceremonial trees, and springs of cultural significance. The RAP grievance mechanism extends to heritage concerns.

20.4.6 Deforestation and community compensation

Project cost estimates include a provision for Deforestation taxes of $1,800 per hectare of land cleared for development. A separate allowance has been made for compensation for crops, loss of land and resettlement costs paid to affected communities, based on an established rate per hectare.

20.4.7 Key Issues raised

The ESIA report includes a summary of the principal issues identified for each aspect, as well as the frequency with which these concerns were expressed.

During the period of public consultation, Kamoa was implementing a large-scale relocation programme. Consequentially, the predominant topics raised by community members throughout the ESIA consultations centered on relocation matters, even among residents of villages not directly impacted by the relocation activities.

Among other concerns most frequently highlighted during consultation sessions were access to water, community health, employment opportunities, education, and land use. These topics consistently emerged as priorities for local stakeholders and were the focus of many discussions throughout the consultation process.

20.4.8 Social risk assessment

The ESIA's risk assessment framework rates physical and economic displacement as the highest potential impact, expected to be mitigated through RAPs and livelihood restoration. Other risks - such as pressure on land, basic services, and potential for conflict - are managed through the ESMP, ongoing engagement, and grievance mechanisms. Vulnerable groups (women, children, elderly, disabled) are specifically considered in all planning.

No unmitigable social risks were identified however the significance of "Increased Physical and Economic Displacement" remains rated as extreme after mitigation considerations.

20.5 Mine closure and reclamation

20.5.1 Closure strategy

A Mine closure and reclamation plan (MCRP) for Kamoa was completed in 2023 by OMI Solutions and is framed around progressive rehabilitation during operations and a risk-informed approach to final closure. The overall intent is to leave all disturbed areas physically safe, geotechnically stable, geochemically non-polluting, and capable of sustaining an agreed post-mining land use. The closure objectives described in the most recent closure plan (Kamoa, 2025) include:

- Biophysical objectives: including the stabilization or removal of surface infrastructure and residues, prevention of adverse effects on environmental quality, reinstating land capability, creation of a self-sustaining landscape, and the support of re-establishment of native vegetation over time.
- Socio-economic objectives: such as stakeholder engagement on post-mining land use options, management of closure and post-closure expectations, the support of sustainable community benefit, and the reduction of community dependence on the mine over the life-of-mine.

The closure strategy is also explicitly integrated into life-of-mine planning through concurrent or progressive reclamation actions that reduce residual closure liability and improve final closure outcomes. These include alien invasive species removal, rehabilitation of watercourses with appropriate vegetation, ongoing water-quality management, and careful management of stripped organic/topsoil materials for later re-use in rehabilitation. Additionally, the closure plan recommends systematic measures to improve closure readiness over time, including maintenance of a life-of-mine material balance to confirm the availability of suitable rehabilitation materials, conducting rehabilitation trials during operations to test growth media and revegetation options, and monitoring and maintenance of rehabilitated areas to reduce post-closure vegetation failure risks.

A critical strategic element is the alignment of closure outcomes with post-mining land use expectations and stakeholder priorities. The current plan has no definitive post-mining land uses assessed in detail yet, and it is noted that the costing basis assumes demolition of surface infrastructure and reinstatement earthworks (e.g., ripping, topsoil replacement, revegetation), assuming an expected post-closure landscape consistent with bushveld/grassland type outcomes. Selected infrastructure (e.g., community-related facilities) could potentially remain in a post-mining scenario where useful infrastructure is preserved, subject to stakeholder agreement and transfer arrangements. Accordingly, continued stakeholder engagement on post-mining land use planning and on success criteria is identified as a necessary input to refine closure actions and reduce social closure risk.

20.5.2 Post-closure monitoring

Post-closure monitoring is a core element of the closure strategy and is intended to verify that closure measures achieve defined completion and success criteria and that residual risks remain controlled. Closure management requirements emphasize that monitoring and evaluation are undertaken to collect data to verify completion criteria and other closure obligations, and that monitoring should confirm that post-mining land uses function as intended. Monitoring is currently planned for a minimum of 10 years post-closure and is focused on stability and environmental performance of rehabilitated areas including TSFs and waste rock dumps. Performance measurement is proposed against indicators such as groundwater and surface water quality, embankment stability, vegetation establishment, and erosion control. This focus is consistent with the MRCP's recommendations to continue surface and groundwater monitoring to assess potential impacts from mine-affected water on the receiving environment.

The monitoring programme follows a risk-based design as per DRC legislation (both Mining Code and Mining Regulations) and international guidelines and standards (IFC's EHS Guidelines for Mining, 2007). As such, the programme is designed for higher monitoring intensity in early post-closure years with potential step-down once performance criteria are demonstrably met, while retaining the ability to re-intensify monitoring if trigger thresholds are exceeded or unanticipated trends emerge.

The financial provision includes explicit post-closure monitoring and aftercare budget items, demonstrating that monitoring is planned and funded as a multi-year activity. In addition, internal closure management requirements note that closure cost estimates are reviewed on a defined cadence (unscheduled and scheduled estimates reviewed biannually and annually, respectively). This supports an iterative management cycle where monitoring outcomes and inspection findings inform both closure performance evaluation and the updating of closure plans and associated cost provisions as listed in Section 21.

20.6 Qualified Person's Statement

The QP is satisfied that the work undertaken to identify and quantify Environmental, Social, and Community compliance requirements and potential impacts has been appropriately addressed. Provisions to prevent, manage and mitigate these impacts have been adequately developed and funded.

21 Capital and operating costs

KCSA capital and operating costs have been estimated through analysis conducted to at least a prefeasibility study (PFS) standard, drawing on contractual and quoted firm estimates, as well as cost data gathered from ongoing operations. Costs reflect current price trends and the production profile in the Mineral Reserve mine plan.

The capital and operating costs have been estimated in a detailed cost model that integrates comprehensive capital cost estimates and physical production plans with detailed cost estimates to generate operating cost estimates.

All figures are based exclusively on the Mineral Reserve mine plan and are expressed in Real US$ as at Q1 2026.

21.1 Capital Costs

The Capital Cost estimate to implement the KCSA Mineral Reserve plan is $17.11 billion including Sustaining Capital. Capital Costs sorted by expenditure category are summarized in Table 21.1 below:

Table 21.1 Mineral Reserve plan Capital Cost

Capital Cost	($) Billion
Mining Development	5.73
Mining Infrastructure	4.69
Mining Mobile Equipment Fleet	1.75
Surface Infrastructure	2.00
Indirects & Other	2.92
Total	**17.11**

For the purposes of this estimate:

- Mining Development covers the cost of underground lateral and vertical development that is considered capitalised, which generally covers all development in waste rock and major life-of-mine access corridors (eg. surface decline to base of mineralisation, primary trucking horizons).
- Mining Infrastructure covers the fitout of all underground lateral and vertical development eg. with services such as electrical, pastefill and all specialised categories chambers (eg. workshops, electrical power substations and distribution, dewatering, refuge bays, underground conveyors etc).
- Mining mobile equipment fleet covers the capital purchase and mid-life capital refurbishment of the dedicated mobile owner mined mobile fleet eg. trucks, loaders, development and production drills, personnel carriers, chargeup vehicle, light vehicle etc over the life of the asset. This cost reflects the assumptions made on the proportion of the work that be completed by owner mining and the proportion completed by engaged contractors using their own fleet.
- Surface Infrastructure covers additional capital works required for the establishment of the new mining operations eg. surface access roads, surface ventilation, electrical and dewatering infrastructure, expansion and future lifts of tailings storage facilities and solar farm facilities. Tailings infrastructure function alone is estimated to be $0.67 billion over the asset life.

No expansions of processing capacity are planned in the Mineral Reserve plan.

21.2 Operating Costs

Operating Costs for the Mineral Reserve mine plan consider the physical activities from operational mining, processing, smelting power, logistics, surface rock handling and general and administration activities.

Table 21.2 describes the estimated operating costs for the Mineral Reserve plan. The costs are presented as a cost per milled tonne unit cost. This unit cost represents the unit cost incurred at steady state production at 17 Mtpa. The value of sulphuric acid production is treated as a credit. The long-term sulphuric acid price is assumed to be US$350/tonne.

Table 21.2 Mineral Reserve plan Operating Cost

Operating Costs	**Steady State ($/t milled) [1)]**
Mining	86.6
Processing	21.2
Smelter	18.0
Logistics charges	12.0
Treatment, Refining & Smelter product charges	3.7
General & Administrative	15.7
Sulphuric Acid Credits	-11.2
C1 Cash Cost	**145.9**

Note: [1] Steady State = Average over 10 consecutive years of steady production at 17 Mtpa.

The power cost component is proportioned between mining, processing and smelting by consumption. It composes $41.9/t milled of the total cost.

21.3 QP comments on Capital and Operating Costs

In the opinion of the QP, the capital and operating cost estimates for the Mineral Reserve plan of the Kamoa-Kakula complex are based on the appropriate level of detailed analysis. The QP has validated that the cost analysis is based on actual costs and appropriate provisions for future cost trends. Appropriate provision has been made in the estimates for the expected operating stages of the operation.

22 Economic analysis

This section is not required as Ivanhoe Mines is a producing issuer, as one of the principal shareholders of Kamoa Copper SA, the operator of the Kamoa-Kakula Mining Complex, the operations are currently in production, and there is no material expansion of current production planned.

An economic analysis of the Kamoa-Kakula Mining Complex has been completed using the Mineral Reserve estimates presented in this Report and the QP has verified that the outcome is a positive cash flow at a $4.50/lb assumed copper sales price which confirms the economic viability of the Mineral Reserves.

23 Adjacent properties

There are no adjacent properties relevant to this Report.

24 Other relevant data and information

There are no additional relevant Data and Information relevant to this Report.

25 Interpretation and conclusions

The May 2025 Seismic event at Kakula is a watershed event for KCSA. The post event analysis has fundamentally reset mine design and operational management philosophy across all deposits. Global stability findings emphasize requirements for company governance and controls of block spans, pillar dimensions, extraction sequencing and backfill timing. The observational approach with defined decision gates before each extraction phase, is a key control. Consistent implementation and maintenance of these controls by operational management should be a focus of senior KCSA management.

The Kamoa-Kakula mining complex is a well-established operation with integrated infrastructure comprising mines, concentrators, smelter, accommodation and infrastructure to support a world leading copper mining and beneficiation organization. The development of the operation has been accelerated and the achievements to date, to establish the business have exceeded those achieved by most peer companies. As the business transitions from a development project to a steady producing operation with a long life, the company should focus on establishing and maintaining the systems, standard practices, and management frameworks that will support reliable integrated production across the life of the operation. The Qualified Person is satisfied that the Kamoa Kalula Copper complex is well established and is able to support the planned production levels over the Mineral Reserve mine life.

The Qualified Person is satisfied that the mine designs and subsequent mining plans are sufficient to demonstrate physically and economically viable mining recovery.

In the opinion of the QP, the metallurgical test work conducted for the Kakula and Kamoa deposits is sufficient for both PFS and feasibility level process design respectively. The samples tested are representative of the target mining area. The comminution characteristics are well established and have consistency across the various testing phases and across the prospective mining areas. Sufficient work has been conducted to design the regrind circuits.

The Qualified Persons are satisfied that the existing and planned access, infrastructure, power and water supply, workforce availability and logistics are reasonably well established to support year-round mining operations.

The Qualified Person considers that geotechnical data confidence varies across the deposits. The operating mines (Kakula, Kamoa 1, Kamoa 2, Kansoko Sud) benefit from substantial underground mapping data. Future deposits (Kamoa 3–6, Kakula West) are supported by wide-spaced drillhole data and have identified gaps in laboratory testing and in situ stress measurement. Recommendations for each deposit include infill geotechnical drilling, expanded laboratory testing programmes (TCS and UTB in particular), in situ stress measurements, and ongoing underground mapping as development advances. Numerical modelling of mine designs is ongoing, and mine plans will be updated as additional data becomes available.

The Qualified Person considers that a perpetual challenge to mining operations will be dewatering ahead of planned underground mining activities. Substantial, proactive analysis, testing and dewatering initiatives are required in the near to medium-term to enable continued operational performance at planned rates.

The QP is satisfied that the work undertaken to identify and quantify Environmental, Social, and Community compliance requirements and potential impacts has been appropriately addressed. Provisions to prevent, manage and mitigate these impacts have been adequately developed and funded.

The Qualified Person has verified that the economic outcome of the Mineral Reserve mine plan is a positive cash flow at a $4.50/lb assumed copper sales price which confirms the economic viability of the Mineral Reserves.

26 Recommendations

26.1 Geology and Mineral Resources

- Drilling has been completed since the database closure dates for the Mineral Resource estimate. While the results of the more recent drilling are not likely to impact on the grade and tonnage of the global estimates, they could enhance the accuracy of local estimates and potentially upgrade Mineral Resource classification.
- It is recommended that Ivanhoe updated the Kamoa and Kakula Mineral Resource estimates periodically as new information is gathered.

26.2 Mining Methods

26.2.1 Geotechnical

- Conduct oriented infill drilling across all mine footprints to improve geotechnical database confidence and progress TLDC ratings to prefeasibility or feasibility levels, particularly at Kamoa 3, 4, 5, 6, and Kakula West.
- Perform additional laboratory testing (TCS and UTB tests) for all rock types (SDT, SSL, SST) across all deposits where current testing is limited. No laboratory testing has been conducted at Kamoa 5 or Kamoa 6 to date.
- Conduct in situ stress measurements at each mine to confirm the extensional normal faulting regime and maximum horizontal stress orientation, using available data from nearby Kamoto (KCC) as a reference only.
- Upgrade the geotechnical database to resolve inconsistencies and gaps in UCS values across different reports, and ensure adequate sampling and geological modelling of key stratigraphic horizons (including Kakula weak breccia and Kamoa KPS units).
- Implement and maintain a Ground Control Management Plan (GCMP) at each operation, including geotechnical hazard registers and Trigger-Action-Response Plans (TARPs).
- Deploy a seismic monitoring system at Kakula and extend seismic monitoring capability to other operations as mining deepens and extraction ratios increase.
- Implement continuous convergence scanning and damage mapping across all operating mines as part of ongoing global stability monitoring.
- Apply Beck Engineering (2025) numerical modelling guidance (calibrated against the Kakula failure event) to all mine designs, updating layouts and extraction sequences; accordingly, continue hydromechanically coupled forward modelling at Kakula.
- Conduct ongoing structural logging and mapping as underground mines are developed to confirm ground support requirements and validate rock mass parameters — particularly at Kamoa 2, 4, 5, and 6 where structural data is currently sparse.
- Conduct site-specific geotechnical investigations at all proposed boxcut/portal sites and ventilation raise locations prior to construction.
- Update ground support standards as new mines are developed, with particular attention to the use of surface support (weld mesh or fibrecrete) in all development backs and shoulders as a minimum standard.
- Implement QA/QC controls on backfill properties (water : cement ratio, slump, particle size distribution) at all paste backfill plants; conduct regular in situ strength testing of cured backfill.

26.2.2 Hydrogeological & Dewatering

- Advance hydrogeological modelling and confirm dewatering designs for all deposits, particularly Kakula West–East where peak inflows of ~4,500 L/s are anticipated.
- Investigate proactive pre-drainage via the drainage horizon in advance of the mining front as well as potential to reduce surface recharge.
- Conduct field stress and permeability measurements within the Roan Sandstone aquifer and along the West Scarp Fault zone to improve confidence in groundwater inflow forecasts.

26.2.3 Mine Design & Planning

- Complete an optimization study to dynamically determine cut-off grades, mining rates, and deposit sequencing across the full KCSA mine and processing infrastructure portfolio.
- Conduct a formal open-pit assessment for near-surface portions of the Kamoa 2 deposit and integrate findings into a combined open-pit/underground extraction strategy.
- Investigate integration of Kakula West–East and Kakula West–West operations to leverage shared infrastructure, access, and ore handling synergies.
- Review and optimize portal/boxcut locations at Kamoa 3 to avoid sterilization of higher-grade ore zones identified in the current layout.
- Investigate access development optimization for Kamoa 3 to better define the economically viable mining footprint and reduce capital development intensity, particularly in 4.5–7 m marginal ore zones.
- Conduct a materials handling trade-off study at Kamoa 3 and Kakula West to evaluate trucking versus conveyor versus shaft-hoist options as mining depth increases.
- At Kamoa 4, collect additional geotechnical and grade data for the Bonanza zone and conduct a detailed comparative mining method assessment prior to the next study phase.
- At Kamoa 5, complete a fully resolved mine design using the adopted 4 m mining height, including re-optimization of pillar dimensions and panel layouts.
- At Kamoa 6, complete infill drilling to better define orebody thickness continuity; use results to confirm or revise mining method selection (R&P vs stoping) and backfill system specifications.
- Review the 80 m access pillar standoff at Kakula West–East in future studies to confirm and optimize this dimension.
- For further optimization at Kakula West–West, conduct geotechnical assessment to refine LHS stope design parameters (level spacing, stope length, minimum mining width) and assess the potential to reduce the 4 m minimum stope width.

26.2.4 Ventilation & Cooling

- Establish a geothermal measurement database at site to replace assumed thermal properties in ventilation/cooling models; update models once sufficient data is available.
- Confirm fan station standardization decision (900 kW trifurcated vs 560 kW bifurcated) across all operations and update capital cost estimates accordingly.

26.2.5 General / Study Advancement

- Implement a formal observational approach to geotechnical hazard management at Kakula during restart, including decision-point gates before advancing each mining stage, informed by the latest geotechnical and geological data.

- Embed lessons learned from the May 2025 Kakula geotechnical event into mine design controls, ground control management plans, and extraction sequence protocols across all KCSA deposits.

26.3 Recovery Methods

26.3.1 Concentrator — Kakula (Project 95)

- Complete and confirm the Kakula P95 modifications — flash flotation cell installation, coarse-fine split cyclone cluster circuit, and reconfigured rougher/scavenger flotation bank.

27 References

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28 QP Certificates

CERTIFICATE OF AUTHOR

I, Karl van Olden, FAusIMM, of Perth, Western Australia, do hereby certify that:

1 I am currently employed as Global Lead – Mining Engineering with AMC Consultants with an office at 1100 Hay St, West Perth, 6005, WA, Australia.

2 This certificate applies to the technical report titled “Kamoa Copper Kamoa-Kakula MRMR Update Technical Report”, with an effective date of 31 March 2026, (the “Technical Report”) prepared for Ivanhoe Mines Ltd (“the Issuer”).

3 I am a graduate of The University of the Witwatersrand in Johannesburg, South Africa (Bachelor of Science Engineering (Mining) degree in 1994). I am a Fellow in good standing of the Australasian Institute of Mining and Metallurgy (AusIMM) (Member # 226473).

4 With a career spanning over 30 years in the mining industry, I have experience gained in South Africa and Australia in both operational and advisory capacities — 16 years in operational roles and 19 years in consulting, including the role of Partner with a global Environmental Consultancy. Since joining AMC Consultants in 2021, I have served as Executive Lead – Consulting and Global Lead - Mining Engineering. My consulting experience includes exposure and responsibility on large assignments for developing and operating client companies across the globe.

5 Throughout my career I have contributed to, managed and sponsored mining studies at scoping, prefeasibility and feasibility levels, covering a wide spectrum of mining operation's functions, including mining and geotechnical engineering, infrastructure, environmental impact, mine closure and mineral economics. In operating environments, I have held operational accountability for mining engineering, geotechnical engineering, infrastructure, community impact and mine closure functions.

6 I have substantial experience as a Qualified Person in the estimation of Mineral Reserves. This experience includes estimation of open pit and underground Mineral Reserves across multiple commodities for developing projects and major mining companies as a practitioner, corporate process custodian and as an independent process compliance reviewer.

7 I hold formal qualifications in Mining Engineering, Mineral Economics (including Minerals Marketing) and Business Administration.

8 I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

9 I have visited the Property in August and December 2025 and January 2026 for 5 days per visit totaling 15 days.

10 I am responsible for portions of Section 1; Sections 2 to 6; Sections15;16; and Sections 18 to 24 and portions of Section 25 and 26. of the Technical Report.

11 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

12 I have not had prior involvement with the property that is the subject of the Technical Report.

13 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

14 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 31 March 2026

(Original signed)

Karl van Olden (FAusIMM) BSc Eng (mining), GDE, MBA

CERTIFICATE OF AUTHOR

I, Jeremy Charles Witley, Pr. Sci Nat., of Johannesburg, South Africa, do hereby certify that:

1 I am currently employed as Head of Mineral Resources with The MSA Group (Pty) Ltd with an office at Henley House, Greenacres Office Park, Victory Park, Randburg, 2195, South Africa.

2 This certificate applies to the technical report titled “Kamoa Copper Kamoa-Kakula MRMR Update Technical Report”, with an effective date of 31 March 2026, (the “Technical Report”) prepared for Ivanhoe Mines Ltd.(“the Issuer”).

3 I graduated with a BSc (Hons) degree in Mining Geology from the University of Leicester in 1988. In addition, I obtained a Master of Science degree in Engineering from the University of Witwatersrand in 2015. I am a registered Professional Natural Scientist (Geological Science) with the South African Council for Natural Scientific Professions (SACNASP) and a Fellow of the Geological Society of South Africa. I have 37 years of post graduate experience as a geologist in a wide variety of mineralization styles and have completed Mineral Resource assignments on more than 20 sedimentary hosted copper projects.

4 I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5 I have visited the Kamoa and Kakula from 15–22 August 2022 for 7 days.

6 I am responsible for Sections 1.6 to 1.10, 1.12 to 1.14, 7 to 12 and 14. And Sections of 25 and 26 of the Technical Report.

7 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

8 I have had prior involvement with the property that is the subject of the Technical Report, being a Qualified person for the Technical Report titled “Kamoa-Kakula Integrated Development Plan 2023”

9 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 31 March 2026

(Original signed by)

Jeremy Witley

Pr. Sci Nat

CERTIFICATE OF AUTHOR

I, Stephen Amos, BSc (Hons), MSc (Eng), FSAIMM (703500), of Johannesburg, Gauteng, do hereby certify that:

1 I am currently employed as Executive Vice President, Projects at Ivanhoe Mines with an office at 82 Maude Street, Second Floor, Sandton, 2146, South Africa.

2 This certificate applies to the technical report titled “Kamoa Copper Kamoa-Kakula MRMR Update Technical Report”, with an effective date of 31 March 2026, (the “Technical Report”) prepared for Ivanhoe Mines Ltd. (“the Issuer”).

3 I am a graduate of the University of the Witwatersrand. I am a member in good standing of the Southern African Institute of Mining and Metallurgy (FSAIMM 703500). I am a practising Project Manager and Process Engineer/Metallurgist and have practised my profession continuously since 1990. I have over 35 years’ experience in the minerals industry. I have been involved in mineral processing, smelting and refining both in studies, operations and projects, from conceptualization to complete project execution. Major commodities including copper, cobalt, gold, uranium, nickel, zinc and platinum group metals (PGMs).

4 I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5 I have visited the Kamoa and Kakula multiple times throughout 2025 including the most recent visit from 4 – 9 August 2025 for 6 days.

6 I am responsible for Sections 1.17.4, 17.4 of the Technical Report.

7 I am not independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

8 I have had prior involvement with the property and am not independent of the issuer that is the subject of the Technical Report.

9 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 31 March 2026

(Original signed by)

Stephen Amos

BSc (Hons), MSc (Eng), FSAIMM (703500)

CERTIFICATE OF AUTHOR

I, Tony Nyakudarika, Professional Engineer, of Johannesburg, South Africa, do hereby certify that:

1 I am currently employed as a Process Consultant with DRA with an office at Building 33 Woodlands Office Park, 20 Woodlands Drive 20 Woodlands Drive Woodlands, Sandton, 2080, South Africa.

2 This certificate applies to the technical report titled “Kamoa-Kakula MRMR Update Technical Report”, with an effective date of 31 March 2026, (the “Technical Report”) prepared for Ivanhoe Limited.(“the Issuer”).

3 I am a graduate of Loughborough Univeristy of Technology in Loughborough, England (Bachelors Degree (Hons. Chemical Engineering) + Diploma in Industrial Studies in 1981). I am a member in good standing of the Association of Engineering Council of South Africa (License #20140392), and a member of the South African Institute of Mining and Metallurgy. I have experience in formulating I have experience in formulating metallurgical test work programmes, flowsheet development, design and operation of metallurgical process plants.

4 I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5 I have visited the Kamoa Copper site from 23 to 28 September 2025 for 5 days.

6 I am responsible for Sections 13, 17, and parts of and 21 of the Technical Report.

7 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

8 I have not had prior involvement with the property that is the subject of the Technical Report.

9 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 31 March 2026

(Original signed by)

Tony Nyakudarika

BSc (Hons) Chem. Eng.(Pr.Eng)

CERTIFICATE OF AUTHOR

I, Andrew Savvas, Pr Eng, CPEng MIEAust., of Johannesburg, Gauteng, South Africa do hereby certify that:

1 I am currently employed as a Director with Epoch Resources (Pty) Ltd with an office at 8 Viscount Road, Block B, Bedfordview, Germiston, Gauteng, South Africa.

2 This certificate applies to the technical report titled "NAME OF REPORT", with an effective date of DD MONTH 2016, (the "Technical Report") prepared for COMPANY.("the Issuer").

3 I graduated from the University of the Witwatersrand, Johannesburg, South Africa with a Bachelor of Science in Civil Engineering in 1993 and a Master of Science in Engineering (Civil) in 1996. I am registered as a Professional Engineer with the Engineering Council of South African (ECSA Pr Eng no. 202403386) and I am a Member of the Institute of Materials, Minerals and Mining (MIMMM no. 691401). I have practised my profession in the mining related sector since 1996 and have specialised in the field of mine residue disposal. I have been involved in mine residue disposal related projects in South Africa as well as Tanzania, Ghana, Lesotho, Namibia, Zimbabwe, Democratic Republic of the Congo, Liberia, Botswana, Niger, Cyprus and Brazil. My working experience has focused on the design of tailings storage facilities with specific emphasis on construction and operational issues, seepage and slope stability analyses, flow slide assessments, probabilistic water balance assessments, and surface water management. As a result of my qualifications and experience, I am a Qualified Person as defined in National Instrument 43-101.

4 I have visited the PROPERTY site from 2 March to 5 March, 2026 for 3 days.

5 I am responsible for Section 18.10 (Tailings Storage Facilities) of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7 I have not had prior involvement with the property that is the subject of the Technical Report.

8 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 31 March 2026

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