Independent Mining Consultants

KAMOA COPPER SA

Kamoa-Kakula Project

Kamoa-Kakula 2018 Resource Update

March 2018

Job No. 17001

IMPORTANT NOTICE

This notice is an integral component of the Kamoa-Kakula 2018 Resource Update and should be read in its entirety and must accompany every copy made of the Technical Report. The Kamoa-Kakula 2018 Resource Update has been prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects.

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The Report Contributors have used their experience and industry expertise to produce the estimates and approximations in the Kamoa-Kakula 2018 Resource Update. Where the Report Contributors have made those estimates and approximations, they are subject to qualifications and assumptions and it should also be noted that all estimates and approximations contained in the Kamoa-Kakula 2018 Resource Update will be prone to fluctuations with time and changing industry circumstances.

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Title Page

Project Name:	Kamoa-Kakula Project
Title:	Kamoa-Kakula 2018 Resource Update
Location:	Lualaba Province Democratic Republic of the Congo
Effective Date of Technical Report:	23 March 2018
Effective Date of Mineral Resources:	Kamoa: 27 November 2017 Kakula: 23 February 2018
Effective Date of Drilling Database:	Kamoa: 23 November 2015 Kakula: 23 February 2018
Effective Date of Mineral Reserves:	28 November 2017

Qualified Persons:

- Bernard Peters, B. Eng. (Mining), FAusIMM (201743), employed by OreWin as Technical Director - Mining was responsible for: Sections 1.1, 1.3, 1.4, 1.12, 1.13, 1.13.5, 1.14.1 to 1.14.3, 1.14.5, 1.15.1, 1.15.3, 1.16.1; Section 2; Section 3; Section 4, Section 5; Section 10.8, Section 16.3; Section 19; Section 20; Sections 21.1, 21.6 to 21.10; Section 22; Section 23; Sections 24.1 to 24.5, 24.7.8, 24.8; Sections 25.1, 25.3, 25.5; Section 26.1; Section 27.
- Dr. Harry Parker, SME Registered Member (2460450), Technical Director, Amec Foster Wheeler a division of Wood plc was responsible for: Sections 1.2, 1.5 to 1.9, 1.11, 1.15.2, 1.16.2; Section 2; Section 3; Section 6; Section 7; Section 8; Section 9; Sections 10.1 to 10.6, 10.10 to 10.12; Sections 11.1 to 11.3, 11.5 to 11.12; Section 12, Section 14; Section 25.2; Section 26.2; Section 27.
- Gordon Seibel, SME Registered Member (2894840), Principal Geologist, Amec Foster Wheeler a division of Wood plc was responsible for: Sections 1.2, 1.5 to 1.9, 1.11, 1.15.2, 1.16.2; Section 2; Section 3; Section 6; Section 7; Section 8; Section 9; Sections 10.1 to 10.6, 10.10 to 10.12; Sections 11.1 to 11.3, 11.5 to 11.12; Section 12, Section 14; Section 25.2; Section 26.2; Section 27.
- William Joughin, FSAIMM (55634), employed by SRK Consulting (South Africa) (Pty) Ltd as Principal Consultant, was responsible for: Section 2; Section 10.7: Section 16.1.
- Jon Treen P. Eng. (Mining), PEO (90402637), employed by Stantec Consulting International LLC as Mining Business Line Leader, was responsible for: Sections 1.13.1, 1.13.4, 1.15.4; Section 2; Section 3; Section 15; Section 16.2; Section 21.2; Section 25.4; Section 26.3; and Section 27.
- Dean David, FAusIMM(CP) (102351), Technical Director – Process, Amec Foster Wheeler, Mining and Metals, Australia West, was responsible for: Sections 1.10, 1.13.2, 1.13.3, 1.14.4, 1.16.3; Section 2.3 and 2.4; Section 10.9; Section 11.4; Section 13; Section 17; Section 18; Sections 21.3, 21.4, 21.5; Sections 24.6, 24.7; Section 26.4; Section 27.

Signature Page

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	Democratic Republic of the Congo
Effective Date of Technical Report:	23 March 2018

/s B F Peters

Date of Signing:

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

/s H M Parker

Date of Signing:

Harry Parker SME Registered Member (2460450), Technical Director, Amec Foster Wheeler

/s G Seibel

Date of Signing:

Gordon Seibel SME Registered Member (2894840), Principal Geologist, Amec Foster Wheeler

/s W Joughin

Date of Signing:

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

/s J Treen

Jon Treen P. Eng PEO (90402637), Mining Business Line Leader, Stantec Consulting International LLC

/s D David

Date of Signing:

Dean David, FAusIMM(CP) (102351), Technical Director – Process, Amec Foster Wheeler, Mining and Metals, Australia West

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1 SUMMARY

1.1 Introduction

The Kamoa-Kakula 2018 Resource Update has been prepared for Ivanhoe Mines Ltd. (Ivanhoe), to update the Mineral Resources for the Kamoa-Kakula Project with additional Mineral Resource on the Kakula deposit. The Kamoa-Kakula 2018 Resource Update is an independent Technical Report (the Report) for the Kamoa-Kakula Project (the Project) located in the Democratic Republic of the Congo (DRC).

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

The previous Technical Report was the Kamoa-Kakula 2017 Development Plan on the Kamoa-Kakula Project. Ivanhoe has undertaken further mineral resource studies following the Kamoa 2017 PFS that has formed the basis of the Kamoa-Kakula 2018 Resource Update, which summarises the current Ivanhoe development strategy for the Kamoa-Kakula Project. The Kamoa-Kakula 2018 Resource Update provides an update of the Kamoa-Kakula Project Mineral Resource, with the Mineral Reserve from the Kamoa 2017 PFS remaining the same. Aside from the updated Mineral Resource, further study work is currently incomplete and has not determined any results that require material changes to the Kamoa 2017 PFS. The Kamoa-Kakula 2018 Resource Update should be read in this context.

Other than the addition of information relevant to the reporting of the Kakula Resource, the remainder of this report has not been changed from the Kamoa 2017 PFS and remains the most current study work available. Further study work is currently incomplete and has not determined any results that require material changes to the Kamoa 2017 PFS.

1.2 Mineral Resource Estimates

Mineral Resources for the Kakula deposit have been updated to incorporate for the first time Mineral Resources contained in the Kakula West Discovery area and the saddle area between the main Kakula Discovery area and Kakula West shown in Figure 1.1. Mineral Resources and documentation for the Kamoa deposit are the same as those previously reported in the Kamoa-Kakula 2017 Development Plan.

Figure 1.1 Kamoa-Kakula Mining Licence, showing the Kamoa, Kakula And Kakula West Mineral Resource Areas

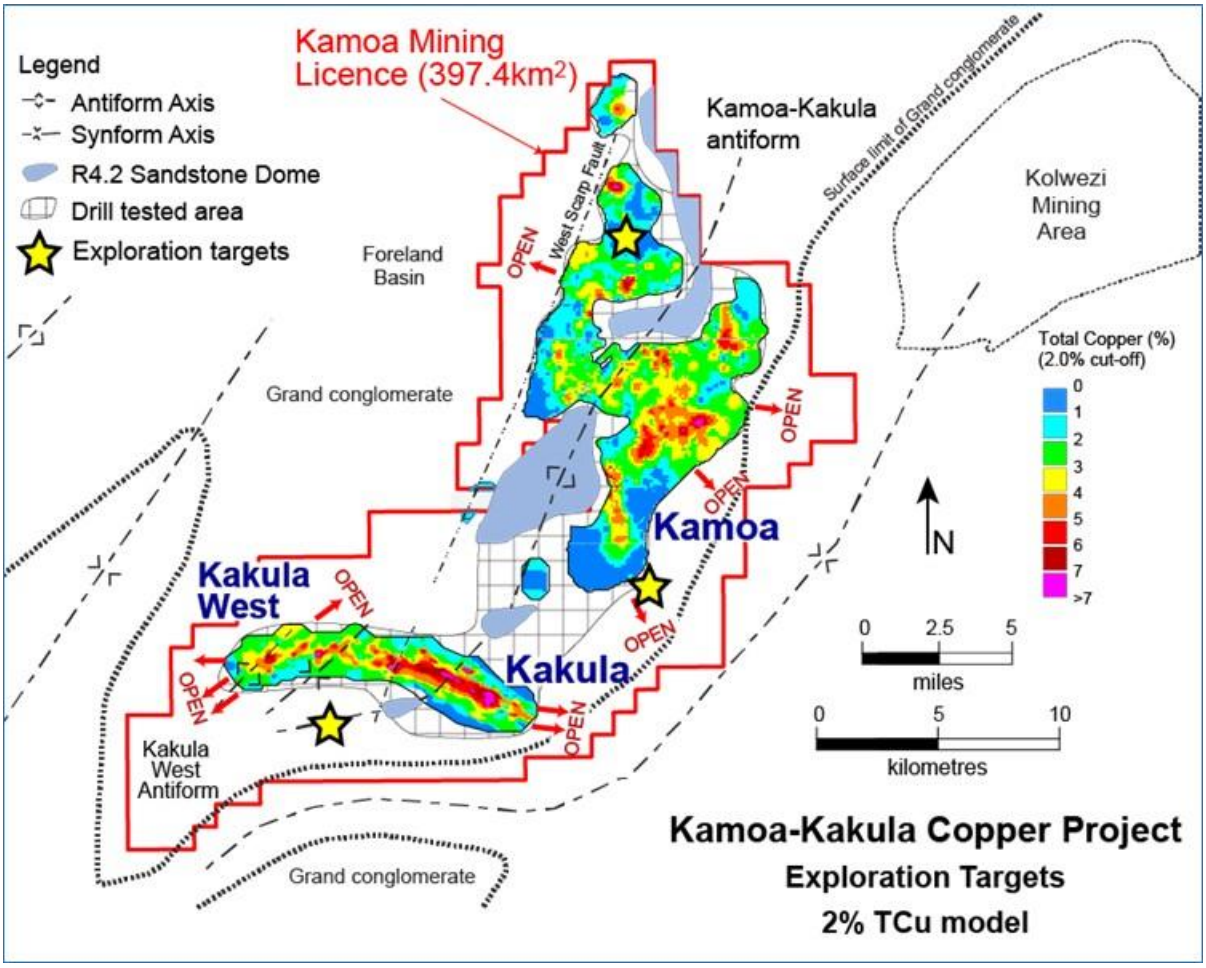

Figure by Kamoa Copper SA, 2018. Kakula West is the Kakula West Discovery; Kakula is the Kakula Discovery

The new estimate increases the total tonnage of Kakula's Indicated Mineral Resources by 50%, at a 3% copper cut-off, along a mineralized strike length of 13.3 kilometres, compared to the previous Kakula resource estimate issued in May 2017 that covered a strike length of 7.7 kilometres.

The Kakula Discovery is situated approximately 10 kilometres southwest of Kamoa's initial Kansoko Mine development (Refer to Figure 1.2).

Figure 1.2 Location Plan Showing the Outlines of the Kamoa-Kakula Indicated and Inferred Mineral Resources

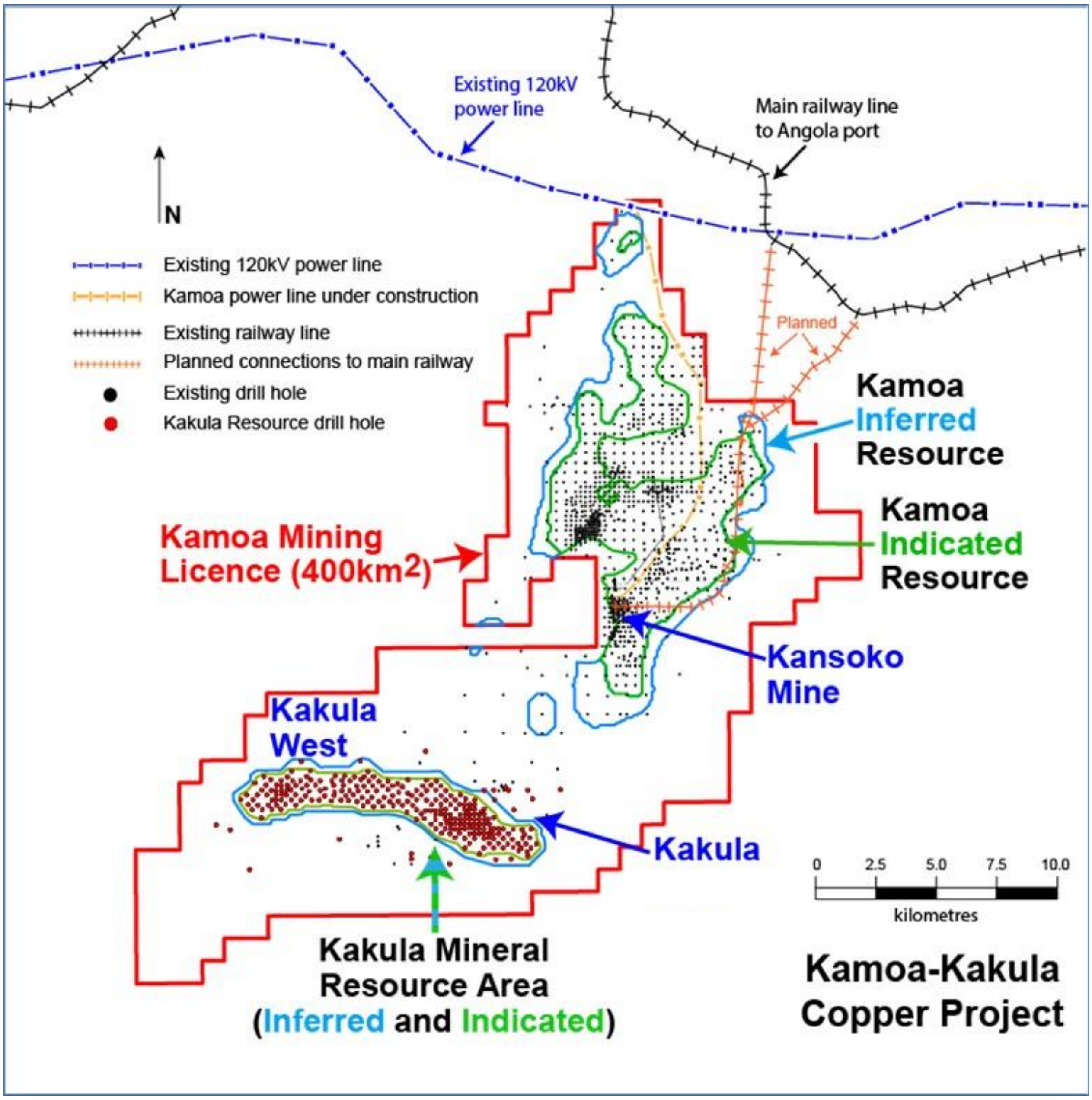

Figure by Kamoa Copper SA, 2018. Kakula West is the Kakula West Discovery; Kakula is the Kakula Discovery

Highlights of the Kakula Mineral Resource update include:

- Indicated Mineral Resources total 585 million tonnes at a grade of 2.92% copper, containing 37.7 billion pounds of copper at a 1% copper cut-off. At a 2% copper cut-off, Indicated Mineral Resources total 330 million tonnes at a 4.07% copper grade, containing 29.6 billion pounds of copper. At a higher cut-off of 3% copper, Indicated Mineral Resources total 174 million tonnes at a grade of 5.62% copper, containing 21.5 billion pounds of copper.
- Inferred Mineral Resources total 113 million tonnes at a grade of 1.90% copper, containing 4.7 billion pounds of copper at a 1% copper cut-off. At a 2% copper cut-off, Inferred Mineral Resources total 44 million tonnes at a 2.59% copper grade, containing 2.5 billion pounds of copper. At a higher cut-off of 3% copper, Inferred Mineral Resources total 9 million tonnes at a grade of 3.66% copper, containing 0.7 billion pounds of copper.
- The average true thickness of the selective mineralized zone (SMZ) at a 1% copper cut-off is 10.1 metres in the Indicated Mineral Resources area and 6.7 metres in the Inferred Mineral Resources area. At a higher 3% copper cut-off, the average true thickness of the SMZ is 4.7 metres in the Indicated Mineral Resources area and 3.3 metres in the Inferred Mineral Resources area.

The Kakula Mineral Resources are defined within a total area of 24.9 square kilometres at a 1% copper cut-off. At the same cut-off grade, the areal extent of Indicated Mineral Resources is 19.4 square kilometres and the areal extent of the Inferred Mineral Resources is 5.5 square kilometres.

The Kakula high-grade mineralized trend remains open in multiple directions. Ivanhoe and Zijin Mining Group Co., Ltd. (Zijin) have been conducting an aggressive drilling program totalling more than 181,500 metres at the Kakula Discovery since April 2016. The program is expected to continue through 2018.

The Kakula Discovery remains open for significant expansion in multiple directions, while the remainder of the southern parts of the Kamoa-Kakula Mining Licence area is virtually untested. Drilling by nine rigs is ongoing at Kakula; more than 25,000 metres have been drilled since the beginning of this year.

The Kakula Mineral Resource estimate followed a modified two-dimensional (2D) modelling approach. Instead of defining a single best selective (or selected) mineralised intercept (SMZ) as previously used at Kamoa, the thickness of the Kakula mineralised zone warranted better vertical definition to capture the very high-grade basal zone. Grade shells were defined at 1%, 2%, and 3% TCu cut-offs. The central 3% SMZ was modelled with a minimum TCu grade of 3%, and a minimum downhole length of 3 m. If the minimum grade criteria could not be met, the highest-grade composite was formed that met the 3 m minimum length. A 1% upper and 2% upper SMZ of variable thickness were modelled above the 3% SMZ, and a 2% lower and 1% lower SMZ of variable thickness were modelled below the 3% SMZ, thus with the 3% SMZ forming five SMZs. An anisotropic search, no top capping, and the use of inverse distance to the power of three (ID3) interpolation, were used to estimate grades; this was done to account for the strong anisotropy and lithological controls on the higher-grade, bottom-loaded style of mineralisation at Kakula.

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

1.2.1 Kamoa-Kakula Mineral Resource Statement

Indicated and Inferred Mineral Resources for Kakula have an effective date of 23 February 2018. Indicated and Inferred Mineral Resources for Kamoa have an effective date of 27 November 2017. Mineral Resources for the Kamoa-Kakula Project are summarised in Table 1.1, and are reported on a 100% basis.

Table 1.2 summarises the Kakula Mineral Resource at a range of cut-off grades. The base case Mineral Resource model reported at a 1.0% TCu cut-off is highlighted in grey.

Table 1.1 Kamoa and Kakula Indicated and Inferred Mineral Resources

Deposit	Category	Tonnes (millions)	Area (Sq. km)	Copper Grade (%)	Vertical Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
Kamoa	Indicated	759	50.7	2.57	5.5	19,500	43.0
	Inferred	202	19.4	1.85	3.8	3,740	8.2
Kakula	Indicated	585	19.4	2.92	10.8	17,100	37.7
	Inferred	113	5.5	1.90	7.3	2,150	4.7
Total Kamoa-Kakula Project	**Indicated**	**1,340**	**70.1**	**2.72**	**6.9**	**36,600**	**80.7**
	Inferred	**315**	**24.9**	**1.87**	**4.6**	**5,890**	**13.0**

1. Ivanhoe's Mineral Resources Manager, George Gilchrist, Professional Natural Scientist (Pr. Sci. Nat) with the South African Council for Natural Scientific Professions (SACNASP), estimated the Mineral Resources under the supervision of Dr. Harry Parker and Gordon Seibel, both Registered Members (RM) of the Society for Mining, Metallurgy and Exploration (SME), who are the Qualified Persons for the Mineral Resource estimate. The effective date of the estimate is 27 November 2017, for Kamoa and 23 February 2018 for Kakula, and the cut-off date for the drill data is 23 November 2015 for Kamoa and 26 January 2018 for Kakula. Mineral Resources are estimated using the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves. Mineral Resources at Kamoa are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
2. Mineral Resources at Kamoa are reported using a total copper (TCu) cut-off grade of 1% TCu and a minimum vertical thickness of 3 m. There are reasonable prospects for eventual economic extraction under assumptions of a copper price of US$3.00/lb,; employment of underground mechanised room-and-pillar and drift-and-fill mining methods,; and that copper concentrates will be produced and sold to a smelter. Mining costs are assumed to be US$27/t, and concentrator, tailings treatment, and general and administrative costs (G&A) are assumed to be US$17/t. Metallurgical recovery for Kamoa is estimated to average 84% (86% for hypogene and 81% for supergene). At a 1% TCu cut-off grade, assumed net smelter returns for 100% of Mineral Resource blocks will cover concentrator, tailings treatment, and G&A costs.
3. Mineral Resources at Kakula are reported using a TCu cut-off grade of 1% TCu and an approximate minimum thickness of 3 m. There are reasonable prospects for eventual economic extraction under assumptions of a copper price of US$3.00/lb, employment of underground, mechanised, room-and-pillar and drift-and-fill mining methods, and that copper concentrates will be produced and sold to a smelter. Mining costs are assumed to be US$42/t, and concentrator, tailings treatment, and G&A costs are assumed to be US$18/t. Metallurgical recovery is assumed to average 85% at the average grade of the Mineral Resource. Ivanhoe is studying reducing mining costs using a controlled convergence room-and-pillar method. At a 1% TCu cut-off grade, assumed net smelter returns for 100% of Mineral Resource blocks will cover concentrator, tailings treatment and G&A costs.
4. Reported Mineral Resources contain no allowances for hangingwall or footwall contact boundary loss and dilution. No mining recovery has been applied.
5. Tonnage and contained-copper tonnes are reported in metric units, contained-copper pounds are reported in imperial units, and grades are reported as percentages
6. Approximate drillhole spacings are 800 m for Inferred Mineral Resources and 400 m for Indicated Mineral Resources.
7. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.

Table 1.2 Sensitivity of Kakula Mineral Resources to Cut-off Grade (base case at 1% Cu is highlighted)

Indicated Mineral Resource						
Cut-off (% Cu)	**Tonnage (Mt)**	**Area (km^2)**	**Copper Grade (%)**	**True Thickness (m)**	**Contained Copper (kt)**	**Contained Copper (billion lbs)**
7.0	41	2.2	8.07	6.3	3,290	7.3
6.0	67	3.6	7.46	6.2	4,970	11.0
5.0	98	5.7	6.82	5.7	6,690	14.7
4.0	140	9.0	6.13	5.1	8,560	18.9
3.0	174	12.3	5.62	4.7	9,750	21.5
2.5	208	14.4	5.14	4.8	10,700	23.5
2.0	330	16.6	4.07	6.6	13,400	29.6
1.5	420	18.0	3.55	7.8	14,900	32.9
1.0	585	19.4	2.92	10.1	17,100	37.7
Inferred Mineral Resource						
Cut-off (% Cu)	**Tonnage (Mt)**	**Area (km^2)**	**Copper Grade (%)**	**True Thickness (m)**	**Contained Copper (kt)**	**Contained Copper (billion lbs)**
4.0	2	0.2	4.17	3.3	98	0.2
3.0	9	0.8	3.66	3.3	325	0.7
2.5	17	1.7	3.20	3.2	549	1.2
2.0	44	3.2	2.59	4.3	1,140	2.5
1.5	69	4.5	2.26	5.0	1,560	3.4
1.0	113	5.5	1.90	6.7	2,150	4.7

1. Ivanhoe's Mineral Resources Manager George Gilchrist, a Fellow of the Geology Society of South Africa and Professional Natural Scientist (Pr. Sci. Nat) with the South African Council for Natural Scientific Professions (SACNASP), estimated the Mineral Resources under the supervision of Dr. Harry Parker and Gordon Seibel, both Registered Members (RM) of the Society for Mining, Metallurgy and Exploration (SME), who are the Qualified Persons for the Mineral Resources. The effective date of the estimate is 23 February 2018. Mineral Resources are estimated using the CIM Definition Standards for Mineral Resources and Reserves (2014), and are reported on a 100% basis.
2. Mineral Resources are reported using a total copper (TCu) cut-off grade of 1% TCu and an approximate minimum vertical thickness of 3 m. There are reasonable prospects for eventual economic extraction under assumptions of a copper price of US$3.00/lb, employment of underground, mechanised, room-and-pillar and drift-and-fill mining methods, and that copper concentrates will be produced and sold to a smelter. Mining costs are assumed to be US$42/t. Concentrator, tailings treatment and general and administrative (G&A) costs are assumed to be US$18/t. Metallurgical recovery is assumed to average 85%. Ivanhoe is studying reducing mining costs using a controlled convergence room-and-pillar method. At a 1% TCu cut-off grade, 100% of the assumed net smelter returns for Mineral Resource blocks will cover concentrator, tailings treatment and G&A costs.
3. Reported Mineral Resources contain no allowances for hangingwall or footwall contact boundary loss and dilution. No mining recovery has been applied.
4. Tonnage and contained-copper tonnes are reported in metric units, contained-copper pounds are reported in imperial units, and grades are reported as percentages.
5. Rounding as required by reporting guidelines may result in apparent differences between tonnes, grade and contained metal content.

1.2.2 Factors Which May Affect the Resource Estimates

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

- Drill spacing.
  - The drill spacing at the Kamoa and Kakula deposits is insufficient to determine the effects of local faulting on lithology and grade continuity assumptions. Local faulting could disrupt the productivity of a highly-mechanised operation. In addition, the amount of contact dilution related to local undulations in the SMZ has yet to be determined for both deposits. Ivanhoe plans to study these risks with the declines currently in progress at Kamoa. A similar decline is being developed to provide access to the Kakula deposit.
  - Delineation drill programs at the Kamoa deposit will have to use a tight (approximately 50 m) spacing to define the boundaries of mosaic pieces (areas of similar stratigraphic position of SMZs) in order that mine planning can identify and deal with these discontinuities. At the Kakula deposit, the mineralisation appears more continuous compared to Kamoa.
- Assumptions used to generate the data for consideration of reasonable prospects of eventual economic extraction for the Kamoa deposit.
  - Mining recovery could be lower and dilution increased where the dip locally increases on the flanks of the domes. The exploration decline should provide an appropriate trial of the conceptual room-and-pillar mining method on the Kamoa deposit in terms of costs, dilution, and mining recovery. The decline will also provide access to data and metallurgical samples at a bulk scale that cannot be collected at the scale of a drill sample.
- Assumptions used to generate the data for consideration of reasonable prospects of eventual economic extraction for the Kakula deposit.
  - A controlled convergence room-and-pillar technique is being studied which provides the opportunity for reduced costs.
- Metallurgical recovery assumptions at Kamoa.
  - Metallurgical testwork at the Kamoa deposit indicates the need for multiple grinding and flotation steps. Variability testwork has been conducted on only portions of the Kamoa deposit. Additional variability testing is needed to build models relating copper mineralogy to concentrate grade and improve the recovery modelling.
  - A basic model predicting copper recovery from certain supergene mineralisation types has been developed. More variability testing is required to improve this model to the point where it is useful for production planning purposes.
- Metallurgical recovery assumptions at Kakula.
  - Preliminary metallurgical testwork at the Kakula deposit indicates that a high-grade chalcocite-dominant concentrate could be produced at similar or higher recoveries compared to those achieved for Kamoa samples.
  - There is no supergene mineralisation currently identified at Kakula that requires a dedicated recovery model separate from the hypogene recovery prediction method.

- Exploitation of the Kamoa-Kakula Project requires building a greenfields project with attendant infrastructure. Changes in the assumptions as to operating and capital costs associated with the proposed development may affect the base case cut-off grades selected for the Kamoa and Kakula Mineral Resource estimates.
- Commodity prices and exchange rates.
- Cut-off grades.

1.3 Property Description

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

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

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

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

1.4 Mineral and Surface Rights, Royalties, and Agreements

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

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

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

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

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

According to the 2002 Mining Code of the DRC, a company holding a mining exploitation licence is subject to payment of mining royalties. The royalty is due upon the sale of the product and is calculated at 2% of the price received of non-ferrous metals less the costs of transport, analysis concerning quality control of the commercial product for sale, insurance, and marketing costs relating to the sale transaction.

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

On 9 March 2018, Law No. 18/001 amending the 2002 Mining Code was promulgated. As of the date of this report, Law No. 18/001 has not been published in the DRC official gazette and the drafting of mining regulations for the implementation of this new law has not been finalised. Detailed discussions are ongoing with the aim of resolving, in a fair and equitable manner, the mining industry's concerns with the 2018 Mining Code.

As soon as Law No. 18/001 will be published and as there is more clarity on the mining regulations governing the implementation of the 2018 Mining Code, as well as potential adaptations to the 2018 Mining Code, if any, a thorough review will be performed to clarify its implications for the Kamoa-Kakula Project with regard to the commitment made in the share transfer agreement dated 11 November 2016. Information in this report, including economic analysis, is therefore based upon the 2002 Mining Code until such clarity is provided.

1.5 Geology and Mineralisation

The mineralisation identified to date within the Project is typical of sediment-hosted stratiform copper deposits.

The regional geology comprises sedimentary rocks of the 880–500 Ma Katangan basin, which were deposited on Paleoproterozoic composite basement rocks. Katangan strata occur on both sides of the DRC–Zambian border and define a northerly-directed, thin-skinned thrust-and-fold orogenic system, the Lufilian Arc, which resulted from the convergence of the Congo and Kalahari cratons. The metasedimentary rocks that host the Central African Copperbelt mineralisation form a sequence known as the Katanga Supergroup, comprising the Roan, Lower Kundelungu, and Upper Kundelungu Groups. Copper mineralisation can occur at a number of stratigraphic levels within these Groups.

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

At the Kakula deposit, these basal diamictite units have yet to be distinguished, as significant thickening of the diamictite basal units makes correlation with the Kamoa deposit area difficult. Mineralisation is concentrated within a basal siltstone layer occurring just above the Roan (R4.2) contact. From the base of mineralisation upward, the hypogene copper sulphides in the mineralised sequence are zoned with chalcocite (Cu_2S), bornite (Cu_5FeS_4) and chalcopyrite ($CuFeS_2$), with chalcocite being the dominant mineral.

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

1.6 Exploration

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

Work completed by Ivanhoe and third-party contractors on the Project has included geological mapping, geochemical sampling, an airborne geophysical survey, reverse circulation (RC), and core drilling, petrographic studies, Mineral Resource estimation, preliminary engineering studies, a PEA on the Kamoa deposit in 2013 (the Kamoa 2013 PEA), a PFS on the Kamoa deposit in 2016 (the Kamoa 2016 PFS), a PEA in 2016 on the Kakula deposit (the 2016 Kakula PEA), and most recently updated in 2017 the Kamoa-Kakula Development Plan which includes a PFS on Kamoa and a PEA on Kakula, as well as an expanded case for Kamoa-Kakula.

Exploration activities at the Kamoa-Kakula Project are being augmented by ongoing geophysical exploration programmes. A 3,100-kilometre, airborne gravity survey, covering 2,000 square kilometres of the Western Foreland area (including Kamoa-Kakula) was recently completed, and the data are being processed. In addition, seismic equipment, including an AHV-IV 65,000-pound seismic vibrator, has been mobilized to site as part of a plan to run approximately 80 kilometres of seismic traverses across the property, including over the highly prospective Kakula trend.

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

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

1.7 Drilling

The drillhole database used for the Kamoa resource estimate was closed on 23 November 2015, and the drillhole database used for the Kakula resource estimate was closed on 26 January 2018. The resource model for Kamoa was updated as of 27 November 2017. The resource model for Kakula was completed as of 23 February 2018.

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

As at 21 February 2018, there were 1,587 coreholes drilled within the Kamoa-Kakula Project. The November 2017 Kamoa Mineral Resource estimate used 776 drillholes. Included in the 776 drillholes were 16 twin holes (where the spacing between drillholes is <25 m) and six wedge holes. Although a far greater number of holes have been wedged, the wedges have typically been used in their entirety for metallurgical testing, and have thus not been sampled for Mineral Resource estimation purposes. In these cases, only the parent hole is used during Mineral Resource estimation. The current Kakula Mineral Resource estimate discussed in this report used 271 drillhole intercepts (one per hole).

The 540 holes not included in either the November 2017 Kamoa or the February 2018 Kakula estimate were excluded because they were abandoned, unmineralised holes in the dome areas, unsampled metallurgical, civil geotechnical or hydrological drillholes, or were drilled after the closure of the databases. Subsequent to the closure of the database for the Kamoa Mineral Resource estimate (23 November 2015), 30 drillholes have intersected the mineralised zone inside of the modelled area at Kamoa. Standard geological logging methods, sampling conventions, and geological codes have been established for the Project. Geotechnical logging has been undertaken on the majority of the drill cores. Kamoa core recovery in the mineralised units ranges from 0% to 100% and averages 95%. Intervals in the database with 0% recovery likely indicate missing data, as logging does not indicate poor recovery. Visual inspection of the Kamoa core by the Amec Foster Wheeler QPs documented the core recovery to be excellent. All completed holes are surveyed by an independent professional surveyor SD Geomatique using a differential GPS which is accurate to within 20 mm.

The Kakula drillholes have been surveyed by SD Geomatique and E.M.K. Construction SARL. As of 26 January 2018, there were five completed drillholes remaining to be surveyed at Kakula, with two of these (DKMC_DD1228 and DKMC_DD1299) used in the current resource estimate. Visual inspection of the Kakula core by the Amec Foster Wheeler QPs documented the core recovery to be excellent.

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

1.8 Sample Preparation, Analyses, and Security

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 was convenient) above the first presence of mineralisation and/or the mineralised zone was sampled on nominal 1 m intervals to the end of the hole, which is generally 5 m below the Ki1.1/R4.2 contact. Most intervals with visual estimates of >0.1% Cu were sampled at 1.5 m intervals or less.

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

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

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

ALS of Vancouver, British Columbia, acted as the independent check laboratory for drill core samples from part of the 2009 programme and for 2010 through 2017 drilling. ALS is ISO:9001:2008 registered and ISO:17025-accredited.

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

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

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

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

1.9 Data Verification

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

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

1.10 Metallurgical Testwork and Concentrator Design

Between 2010 and 2015 a series of metallurgical testwork programs were completed on drill core samples of known Kamoa copper mineralisation. These investigations focussed on metallurgical characterisation and flowsheet development for the processing of hypogene and supergene copper mineralisation. During this developmental period the known area hosting mineralisation expanded progressively, and this led to major changes to mine schedules and associated processing schedules.

In preparation for the Kamoa 2016 PFS and the increased capacity of the Kamoa 2017 PFS, the Phase 6 samples were selected and the associated metallurgical evaluation was conducted over 2014 and 2015 at Xstrata Process Support (XPS) Laboratories. The Phase 6 samples best represent ores to be processed according to the early years (Years 1 to 15) of the Kamoa 2017 PFS mine schedule.

Bench-scale metallurgical flotation testwork carried out at XPS Consulting and testwork Services laboratories in Falconbridge, Canada on an 8.1% copper head grade sample, achieved copper recoveries of 87.8% and a concentrate grade with 56% copper at 12.5% mass pull. In addition, the arsenic in final concentrate was lower than the previous Kamoa test results. The material tested and the subsequent plant design will accommodate this 8.1% copper feed grade. Both Zijin laboratory in China and XPS laboratories on different composite samples achieved similar recoveries and grades.

Kakula plant feed is expected to be consistent with the Kansoko plant feed on most measures with the major exceptions being grade, rod mill work index (BRWI) and Bond abrasion index. The abrasion index and the copper grade are both favourable but the BRWI is significantly worse. The higher BRWI value has resulted in recommendations to reduce the ball mill feed topsize as much as is practical to minimise or avoid scatting of the primary ball mill.

The project ores do not contain deleterious elements often found in copper concentrates such as arsenic and fluorine. The Kakula ore is especially low in arsenic. As a result, the flotation testwork has consistently generated concentrates that are free of penalty elements.

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

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

Compared to the Kansoko ores, the Kakula deposit has less variability in copper mineralisation, a low and consistent arsenic content and effectively equivalent comminution properties. The flowsheet design work performed on Kansoko ores has been proven as well suited to the Kakula ores, and no major flowsheet changes (apart from those that are needed to accommodate high-grade feed) are envisaged for the project.

The concentrator design incorporates a run-of-mine stockpile, followed by primary and secondary crushing on surface. The crushed material, with a design size distribution of 80% passing (or P_{80}) 9 mm, is fed into a two-stage ball milling circuit for further size reduction to a target grind size P_{80} of 53 micrometres (µm). The milled slurry is subjected to rougher flotation followed by scavenger flotation. The high-grade, or fast-floating rougher concentrate, and medium-grade or slow-floating scavenger concentrate are collected separately. The rougher concentrate is upgraded in two stages of cleaning to produce a high-grade increment to final concentrate. The medium-grade scavenger concentrate and tailings from the two rougher cleaning stages are combined and re-ground to a P_{80} of 10 µm before being cleaned in two stages. The cleaned scavenger concentrate is then combined with the cleaned rougher concentrate to form the final concentrate. The final concentrate is thickened before being pumped to the concentrate filter. Filter cake is then bagged for shipment to market.

1.11 Targets for Further Exploration

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

The area inside the model perimeter surrounding the Indicated and Inferred Mineral Resources at Kamoa is considered an exploration target. The ranges of the Kamoa-Makalu exploration target tonnages and grades are summarised in Table 1.3. Tonnage and grade ranges were estimated using an inverse distance weighting and applying a +/-20% variance to the resulting tonnage and grade estimate.

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

Table 1.3 Tonnage and Grade Ranges for Kamoa-Makalu Exploration Target

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

1.12 Kamoa-Kakula 2017 Development Plan

The Kamoa-Kakula 2018 Resource Update includes restatement of the Kamoa-Kakula 2017 Development Plan which includes the Mineral Reserve on Kansoko from the Kamoa 2017 PFS and the results of the preliminary economic assessment (PEA) from the Kakula 2017 PEA. The Mineral Reserve in the Kamoa 2017 PFS remains valid. Further study work is currently incomplete and has not determined any results that require material changes to the Kamoa 2017 PFS.

The development of the two deposits is at different stages. The Kamoa deposit includes a Mineral Reserve at the Kansoko Mine that is at a prefeasibility study (PFS) level reported here as the Kamoa 2017 PFS (Details are in Sections 15 to 22). Ivanhoe has completed the development of twin declines at the Kansoko Mine to provide access to the Kansoko areas of the Kamoa deposit.

The Kakula deposit was discovered more recently than the Kamoa deposit and consequently the studies of the Kakula deposit are at a preliminary economic assessment (PEA) level reported here as the Kamoa-Kakula 2017 PEA (Details are in Section 24). In November 2017, the development of twin declines to access the Kakula deposit commenced.

The Mineral Resource grade of Kakula is significantly higher than that of Kamoa, and for this reason the analysis of the project considers the separate and combined development of Kakula and Kamoa. Three potential development scenarios have been identified for the Kamoa-Kakula Project:

1. Initial mine development scenario based on Kakula only (Kakula 6 Mtpa PEA). The Kakula 6 Mtpa PEA evaluates the development of a 6 Mtpa underground mine and surface processing complex at the Kakula Deposit as the project's first phase of development.
2. Expanded, two-mine development scenario. The Kamoa-Kakula 12 Mtpa PEA evaluates an integrated, 12 Mtpa, two-stage development, beginning with initial production from a 6 Mtpa underground mine and surface processing complex at the Kakula Mine, to be followed by a subsequent, separate 6 Mtpa underground mining operation at the nearby Kansoko Mine and an associated expansion of the surface processing facilities at Kakula, as well as the construction of a smelter. As the mining at Kakula and Kansoko deposits is completed, mining activities will be extended to include the Kamoa North deposits to the north of Kansoko in order to sustain a 12 Mtpa production rate.
3. Kamoa 2017 PFS. The Kamoa 2017 PFS evaluates the development of the Kansoko Mine as a stand-alone 6 Mtpa underground mine and surface processing complex that would be supplied with ore from the planned development of the Kansoko Sud and Kansoko Centrale areas of the Kamoa Deposit, which were discovered in 2008.

The analysis for each scenario assumes the construction and operation of underground mines, concentrator processing facilities, and associated infrastructure. The base case mining rate for each mine is 6 Mtpa.

The mining locations for the scenarios are shown in Figure 1.3. The production rates and study levels in the Kamoa-Kakula 2017 Development Plan are shown in Table 1.4.

Figure 1.3 Kamoa 2017 PFS and Kamoa-Kakula 2017 PEA Mining Locations

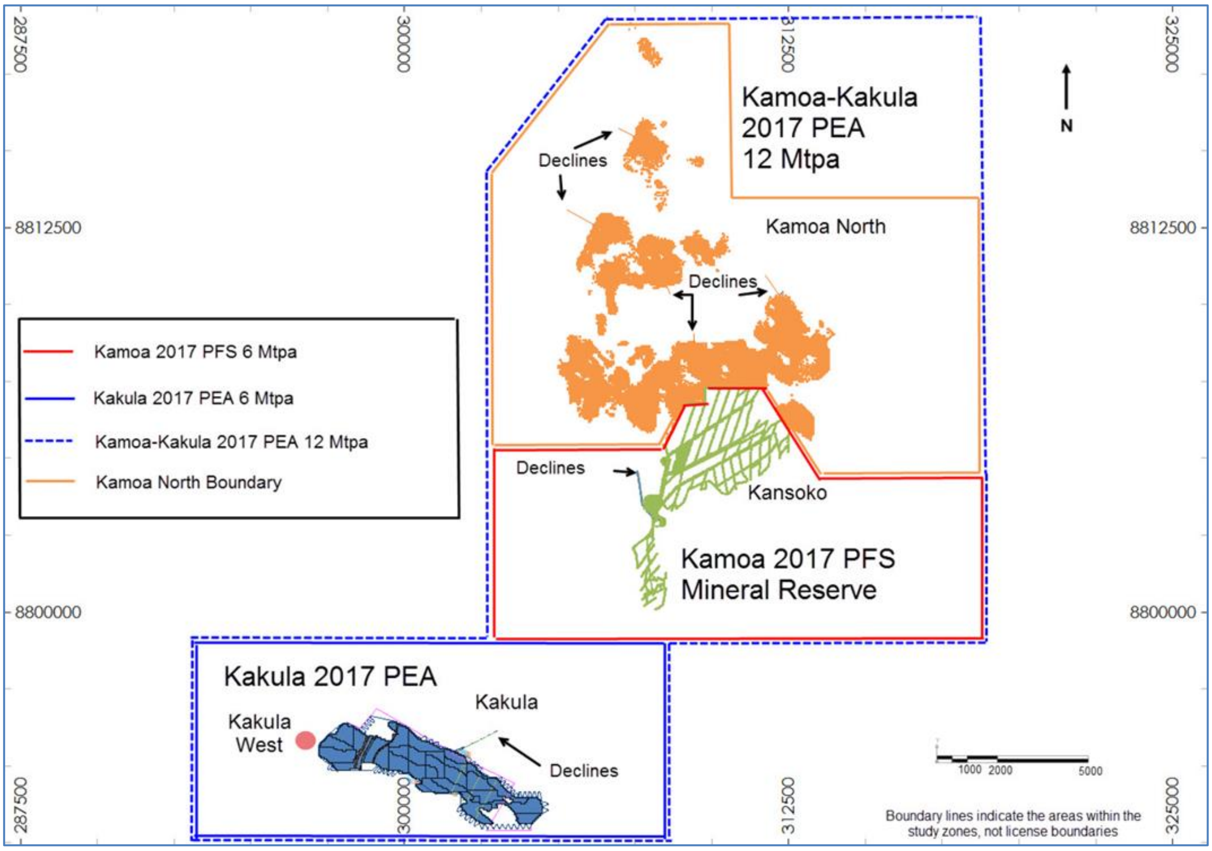

Figure by OreWin, 2017.

Table 1.4 Production Cases Kamoa-Kakula 2017 Development Plan

Study Level	Scenario Total Production Rate (Mtpa)	Plant Production Rate
PFS	6	Kansoko 6 Mtpa Only
PEA	6	Kakula 6 Mtpa Only
PEA	12	Kakula 6 Mtpa, Kansoko 6 Mtpa & Kamoa North 12 Mtpa

The Kamoa-Kakula 2017 PEA includes analysis of the Kakula deposit as a standalone operation and an alternative initial option that could involve a two-phase sequential expansion of production to 12 Mtpa from the proposed Kakula Mine, the Kansoko Mine, and Kamoa North Mines.

The Kakula 2017 PEA is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves – and there is no certainty that the results will be realised. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The potential development scenarios at the Kamoa-Kakula Project include the Kamoa-Kakula 2017 PEA 12 Mtpa development scenario shown in Figure 1.4. The Kakula decline development is followed by the development of the stoping panels and construction of the plant. The initial plant capacity of 3 Mtpa is expanded to 6 Mtpa as the Kansoko Mine and Kakula Mine ramp up. The mines continue to ramp up to 12 Mtpa combined by Year 8. Once the Kansoko and Kakula Mines near the end of their mine life, Kamoa North comes on line to maintain the overall production at 12 Mtpa.

Figure 1.4 Kamoa-Kakula 2017 PEA Long-Term Development Plan

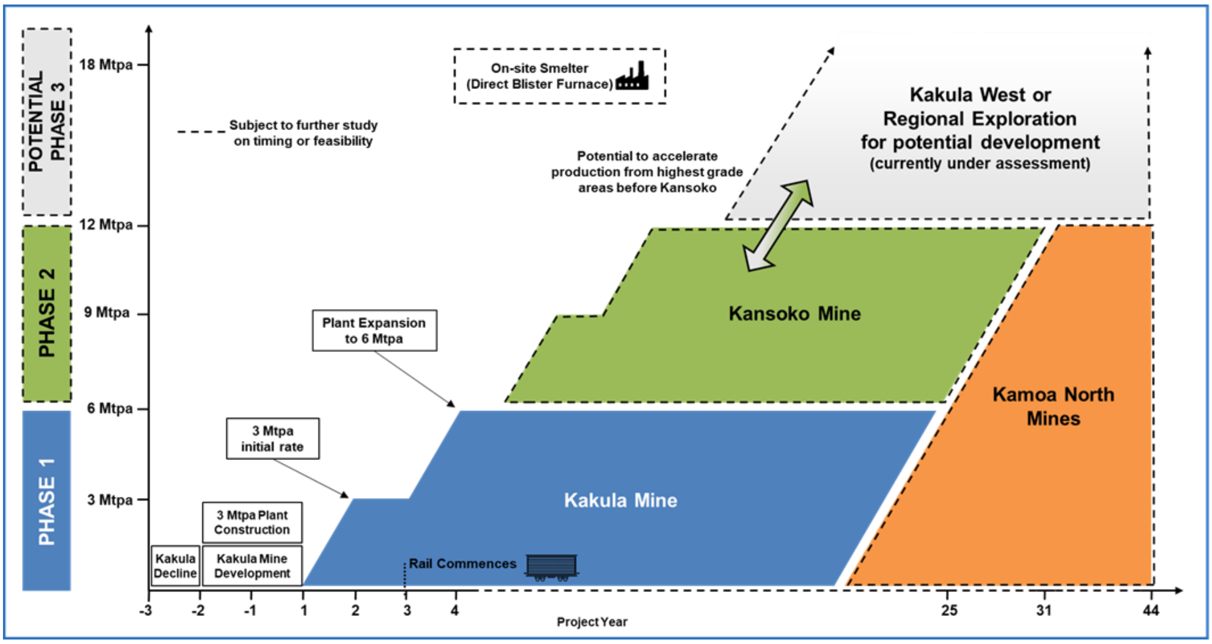

Figure by OreWin, 2017.

1.13 Kamoa 2017 PFS

Ivanhoe has developed twin declines at the Kansoko Mine on the Kansoko areas of the Kamoa deposit. Once in production, one will be a service decline for the transport of personnel and materials into the mine, and the second will be a conveyor decline for rock handling and transport of personnel and materials out of the mine. The Kansoko Mine has a Mineral Reserve that was previously stated in the Kamoa 2016 PFS and was updated in the Kamoa 2017 PFS.

The base case described in the Kamoa 2017 PFS is the construction and operation of an underground mine, concentrator processing facilities, and associated infrastructure. The Kamoa 2017 PFS production is planned to be an average of 6 Mtpa ore over a production period of 26 years.

1.13.1 Kamoa 2017 PFS Mining

A probable Mineral Reserve of approximately 125.2 million tonnes (Mt) grading at 3.81% Cu has been defined in multiple mining zones to support a 6 Mtpa rate over a 26-year mine life. These ore zones occur at depths ranging from approximately 60 m to 1,235 m. Access to the mine will be via twin declines. Main declines and ventilation raises are shown in Figure 1.5. Mining will be performed using the room-and-pillar mining method in the mineralised zone between 60–150 m and controlled convergence room-and-pillar for mineralised zones below 150 m. These methods have been modified from previous studies.

Figure 1.5 Kamoa 2017 PFS Mine Access and Ventilation

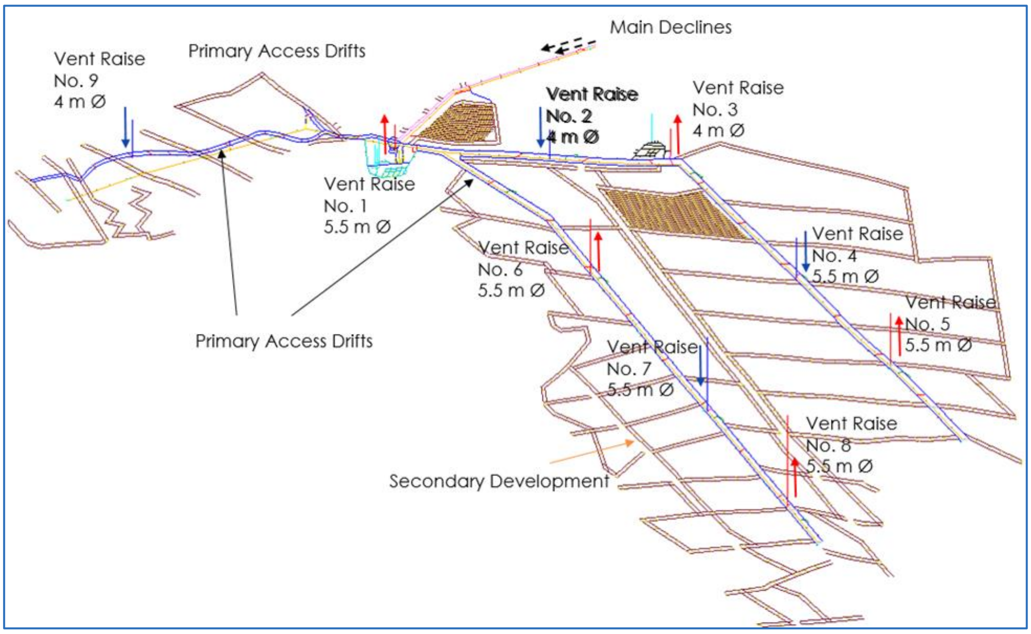

Figure by Stantec, 2017.

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

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

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

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

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

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

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

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

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

Table 1.5 shows life-of-mine (LOM) production summary.

Table 1.5 LOM Production Summary

Production by Mining Method	Mined (kt)	Meters (m)	NSR ($/t)	Cu (%)	AsCu (%)	S (%)	As (%)	Fe (%)	Density (t/m^3)
Ore Development	10,665	114,205	146.20	3.34	0.29	2.40	0.00	6.19	2.90
Room-and-Pillar	3,397	36,743	243.27	5.29	0.55	2.05	0.00	5.28	2.73
Controlled Convergence Room-and-Pillar	111,120	813,559	167.86	3.81	0.32	2.51	0.00	6.17	2.94
Total	**125,182**	**964,507**	**168.06**	**3.81**	**0.32**	**2.49**	**0.00**	**6.14**	**2.93**

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

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

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

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

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

1.13.2 Kamoa 2017 PFS Process

The Kamoa 2017 PFS process plant consists of a 6 Mtpa Run-of-Mine (ROM) concentrator incorporating staged crushing, ball mill grinding and flotation. The output of the process plant is copper concentrate which is sold to external smelters.

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

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

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

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

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

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

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

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

The three regrind mill feed streams (scavenger concentrate, rougher cleaner tails and rougher recleaner tails) are pumped to the regrind feed tank. Regrind circuit feed is pumped to the regrind densifying cyclones. Densifying cyclone overflow reports directly to the regrind product tank and cyclone underflow is fed to the regrind mills. Reground material reports to the regrind product tank. The regrind target P_{80} is10 µm.

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

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

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

In all the scenarios in the Kamoa-Kakula 2017 Development Plan the plant feed is to be treated at the same rate (6 Mtpa) in similar comminution and flotation flowsheets. This flowsheet description is applicable to all scenarios mentioned in Section 1 to save repetition. Minor differences specific to a particular concentrator flowsheet will be described at the point of referencing, if necessary.

1.13.3 Kamoa 2017 PFS Transport

A phased logistics solution is proposed in the Kamoa 2017 PFS. Initially the corridor between southern DRC and Durban in South Africa is viewed as the most attractive and reliable export route. As soon as the railroad between Kolwezi and Dilolo, a town near the DRC-Angolan border, is rehabilitated, production from the Kamoa-Kakula Project is expected to be transported by rail to the port of Lobito in Angola.

1.13.4 Kamoa 2017 PFS Mineral Reserve

The Kamoa-Kakula 2018 Resource Update includes restatement of the Kamoa-Kakula 2017 Development Plan, which includes the Mineral Reserve on Kansoko from the Kamoa 2017 PFS and the results of the preliminary economic assessment (PEA) from the Kakula 2017 PEA. The Mineral Reserve in the Kamoa 2017 PFS remains valid. Further study work is currently incomplete and has not determined any results that require material changes to the Kamoa 2017 PFS.

High-grade copper mineralisation occurs within a sediment-hosted stratiform deposit consisting of relatively thin (generally <30 m and commonly less than 3 m) sulphide-bearing zones, typically consisting of haematite–chalcocite–bornite–chalcopyrite–pyrite. Some native copper is also present in zones of supergene enrichment. Galena and sphalerite may occur with chalcopyrite or between the chalcopyrite and pyrite zones. Minerals are finely disseminated, stratabound, and locally stratiform.

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

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

Tonnes and grades were calculated for panels, and allowances for unplanned dilution and mining recovery were applied to calculate the Probable Ore Reserves. The total Mineral Reserves for the Kamoa 2017 PFS are summarised in Table 1.6.

Table 1.6 Kamoa 2017 PFS Mineral Reserve

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

1. Effective date of the Mineral Reserve is 28 November 2017.
2. The copper price used for calculating the financial analysis is long-term copper at US$3.00/lb. The analysis has been calculated with assumptions for smelter refining and treatment charges, deductions and payment terms, concentrate transport, metallurgical recoveries and royalties.
3. For mine planning, the copper price used to calculate block model Net Smelter Returns was US$3.00/lb.
4. An elevated cut-off of US$100.00/t NSR was used to define the stoping blocks. A cut-off of US$80.00/t NSR was used to define ore and waste for the mine plan.
5. Indicated Mineral Resources were used to report Probable Mineral Reserves.
6. The Mineral Reserves reported above are not additive to the Mineral Resources.

1.13.5 Kamoa 2017 PFS Results

The base case described in the Kamoa 2017 PFS is the construction and operation of a stand-alone underground mine, concentrator processing facilities, and associated infrastructure. The base case mining rate and concentrator feed capacity is 6 Mtpa. This refines the findings of the Kamoa March 2016 PFS, which envisaged a production rate of 3 Mtpa. The PFS is based entirely on the Kamoa 2017 PFS Mineral Reserve, details of which are shown in Table 1.6.

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

The economic analysis uses a long-term price assumption of US$3.00/lb of copper and returns an after-tax NPV at an 8% discount rate of US$2.1 billion, an increase of 110% compared to the after-tax NPV8% of US$986 million that was projected in the Kamoa 2016 PFS. It has an after-tax IRR of 24.2% and a payback period of 5.0 years. The life-of-mine average mine site cash cost is US$0.64/lb of copper.

The estimated initial capital cost, including contingency, is US$1.0 billion. The capital expenditure for off-site power, which is included in the initial capital cost, includes a US$71 million advance payment to the DRC state-owned electricity company, SNEL, to upgrade two hydropower plants (Koni and Mwadingusha) to provide the Kamoa-Kakula Project with access to clean electricity for its planned operations. Mwadingusha is being upgraded first. The work is being led by Stucky Ltd., of Switzerland; the advance payment will be recovered through a reduction in the power tariff once Kamoa is in operation.

The key results of the Kamoa 2017 PFS are summarised in Table 1.7.

Table 1.7 Kamoa 2017 PFS Summary

Item	Unit	Total
Total Processed		
Quantity Milled	kt	125,182
Copper Feed Grade	%	3.81
Total Concentrate Produced		
Copper Concentrate Produced	kt (dry)	11,405
Copper Recovery	%	87.52
Copper Concentrate Grade	%	36.63
Contained Metal in Concentrate	Mlb	9,211
Contained Metal in Concentrate	kt	4,178
Peak Annual Contained Metal in Concentrate	kt	245
10 Year Average		
Copper Concentrate Produced	kt (dry)	487
Contained Metal in Concentrate	kt	178
Mine Site Cash Cost	US$/lb	0.57
Total Cash Cost	US$/lb	1.44
Key Financial Results		
Peak Funding	US$M	1,070
Initial Capital Cost	US$M	1,004
Expansion Capital Cost	US$M	348
Sustaining Capital Cost	US$M	1,334
LOM Average Mine Site Cash Cost	US$/lb Cu	0.64
LOM Average Total Cash Cost	US$/lb Cu	1.51
Site Operating Cost	US$/t Milled	45.21
After-Tax NPV8%	US$M	2,063
After-Tax IRR	%	24.2
Project Payback Period	Years	5.0
Initial Project Life	Years	26

1.14 Kamoa-Kakula 2017 PEA

The Kamoa-Kakula 2018 Resource Update includes restatement of the Kamoa-Kakula 2017 Development Plan which includes the Mineral Reserve on Kansoko from the Kamoa 2017 PFS and the results of the preliminary economic assessment (PEA) from the Kakula 2017 PEA. The results of this work remains valid. Further study work is currently incomplete and has not determined any results that require material changes to the Kamoa 2017 PFS.

The Kamoa-Kakula 2017 PEA was prepared to provide two initial scenarios for development of the high-grade copper deposits at the Kamoa-Kakula Project on the Central African Copperbelt, west of the DRC's Katanga mining region.

The Kamoa-Kakula 2017 PEA includes analysis of the Kakula deposit as a standalone operation and an alternative initial option that could involve a two-phase sequential expansion of production to 12 Mtpa from the proposed Kakula Mine, the Kansoko Mine and Kamoa North Mines.

The two PEA production scenarios are:

- Kakula 6 Mtpa PEA (includes the Kakula Mine only).
- Kamoa-Kakula 12 Mtpa PEA.

Both the Kamoa-Kakula 2017 PEA scenarios assume initial production from Kakula. The Kakula decline development is followed by the development of the stoping panels and construction of the plant. The Kakula 6 Mtpa PEA initial plant capacity is 3 Mtpa and then expanded to 6 Mtpa.

In the Kamoa-Kakula 12 Mtpa PEA scenario once Kakula reaches full production of 6 Mtpa the Kansoko Mine commences and the plant at Kakula is expanded total production rate reaches 12 Mtpa after approximately nine years. The Kamoa-Kakula 12 Mtpa PEA scenario also includes an on-site smelter to produce blister copper at the mine site, which commences production as the 12 Mtpa rate is reached. Once the Kansoko and Kakula Mines near the end of their mine life, Kamoa North comes on line to maintain the overall production rate at 12 Mtpa.

The potential development scenarios at Kamoa-Kakula Project including the Kamoa-Kakula 12 Mtpa PEA development scenario is shown in Figure 1.6 and an overview of deposits included within Kamoa-Kakula 2017 PEA (6 Mtpa and 12 Mtpa cases) and Kamoa 2017 PFS (6 Mtpa) is shown in Figure 1.7.

Figure 1.6 Kamoa-Kakula 2017 PEA Long-Term Development Plan

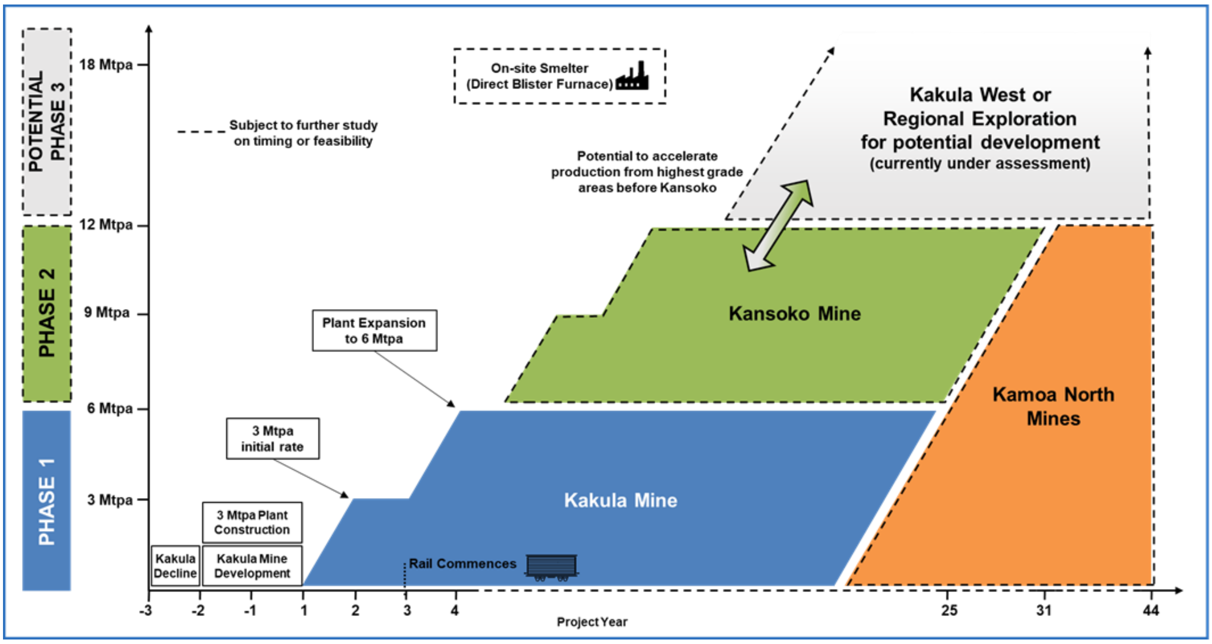

Figure by OreWin, 2017.

Figure 1.7 Kamoa 2017 PFS and Kamoa-Kakula 2017 PEA Mining Locations

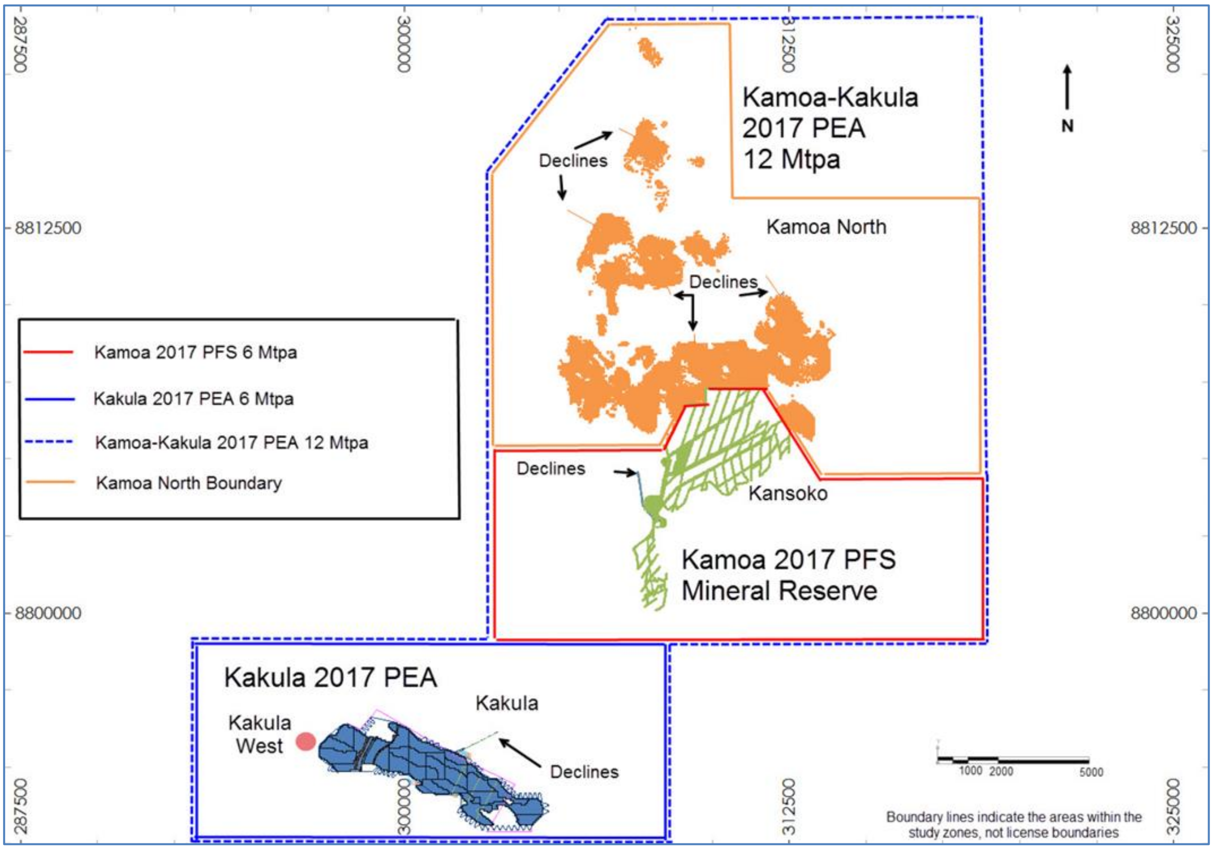

Figure by OreWin, 2017.

The first scenario of the Kamoa-Kakula 2017 PEA, the Kakula 6 Mtpa PEA, represents the initial phase of the Kakula development. This option envisages an average annual production rate of 284 kt of copper at a mine site cash cost of US$0.51/lb copper and total cash cost of US$1.14/lb copper for the first five years of operations, and annual copper production of up to 320 kt by Year 9. The pre-production capital cost of US$1.2 billion for this option would result in an after-tax net present value at an 8% discount rate (NPV8%) of US$4.2 billion.

The Kamoa-Kakula 2017 PEA is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves – and there is no certainty that the results will be realised. Mineral Resources do not have demonstrated economic viability and are not Mineral Reserves.

The Kamoa-Kakula 12 Mtpa PEA scenario envisages US$1.2 billion in initial capital costs. Future expansion at the Kansoko Mine and subsequent extensions could be funded by cash flows from the Kakula Mine, resulting in an after-tax net present value at an 8% discount rate (NPV8%) of US$7.2 billion and an internal rate of return of 33%. Under this approach, the Kamoa-Kakula 12 Mtpa PEA also includes the construction of a direct-to-blister flash copper smelter with a capacity of 690,000 tonnes of copper concentrate per annum to be funded from internal cash flows. This would be completed in Year five of operations, achieving significant savings in treatment charges and transportation costs.

The Kamoa-Kakula 12 Mtpa PEA scenario has an average annual production of 370,000 tonnes of copper at a total cash cost of US$1.02/lb copper during the first 10-years of operations and annual production of 542,000 tonnes by Year Nine. At this future production rate, Kamoa-Kakula would rank among the world's five largest copper mines. The results of the two PEA scenarios are summarised in Table 1.8.

A plan showing the locations of the mines and key infrastructure for Kakula and Kansoko mines is shown in Figure 1.8.

The Kamoa-Kakula 2018 Resource Update includes economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability. The results of the Kamoa-Kakula 2018 Resource Update represent forward looking information. The forward-looking information includes metal price assumptions, cash flow forecasts, projected capital and operating costs, metal recoveries, mine life and production rates, and other assumptions used in the Kamoa-Kakula 2018 Resource Update. Readers are cautioned that actual results may vary from those presented. The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially are presented in the body of this report under each relevant section.

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

Table 1.8 Kamoa-Kakula 2017 PEA Scenarios Summary

Item	Unit	Kakula 6 Mtpa PEA	Kamoa-Kakula 12 Mtpa PEA
Total Processed			
Quantity Milled	kt	108,422	444,276
Copper Feed Grade	%	5.48	3.79
Copper Concentrate Produced	kt (dry)	9,400	34,206
Copper Concentrate - External Smelter	kt (dry)	9,400	9,744
Copper Concentrate - Internal Smelter	kt (dry)	–	24,461
Copper Recovery	%	86.86	85.97
Copper Concentrate Grade	%	54.94	42.30
Cont. Metal in Conc. - External Smelter	Mlb	11,385	10,627
Cont. Metal in Conc. - External Smelter	kt	5,164	4,820
Cont. Metal in Blister - Internal Smelter	Mlb	–	20,955
Cont. Metal in Blister - Internal Smelter	kt	–	9,505
Peak Annual Contained Metal in Concentrate	kt	385	542
10 Year Average			
Copper Feed Grade	%	6.42	5.72
Copper Concentrate Produced	kt (dry)	517	758
Cont. Metal in Conc. - External Smelter	kt	284	188
Cont. Metal in Blister - Internal Smelter	kt	–	182
Mine Site Cash Cost (Including Smelter)	US$/lb	0.51	0.63
Total Cash Cost (After Credits)	US$/lb	1.14	1.02
Key Financial Results			
Peak Funding	US$M	1,135	1,139
Initial Capital Cost	US$M	1,231	1,235
Expansion Capital Cost	US$M	318	3,647
Sustaining Capital Cost	US$M	1,443	5,133
LOM Avg. Mine Site Cash Cost (Including Smelter)	US$/lb	0.60	0.91
LOM Avg. Total Cash Costs (After Credits)	US$/lb	1.23	1.20
Site Operating Cost	US$/t Milled	61.49	64.17
After-Tax NPV8%	US$M	4,243	7,179
After-Tax IRR	%	36.2	33.0
Project Payback	Years	3.1	4.7
Initial Project Life	Years	24	44

Figure 1.8 Kamoa-Kakula 2017 Development Plan Site Plan

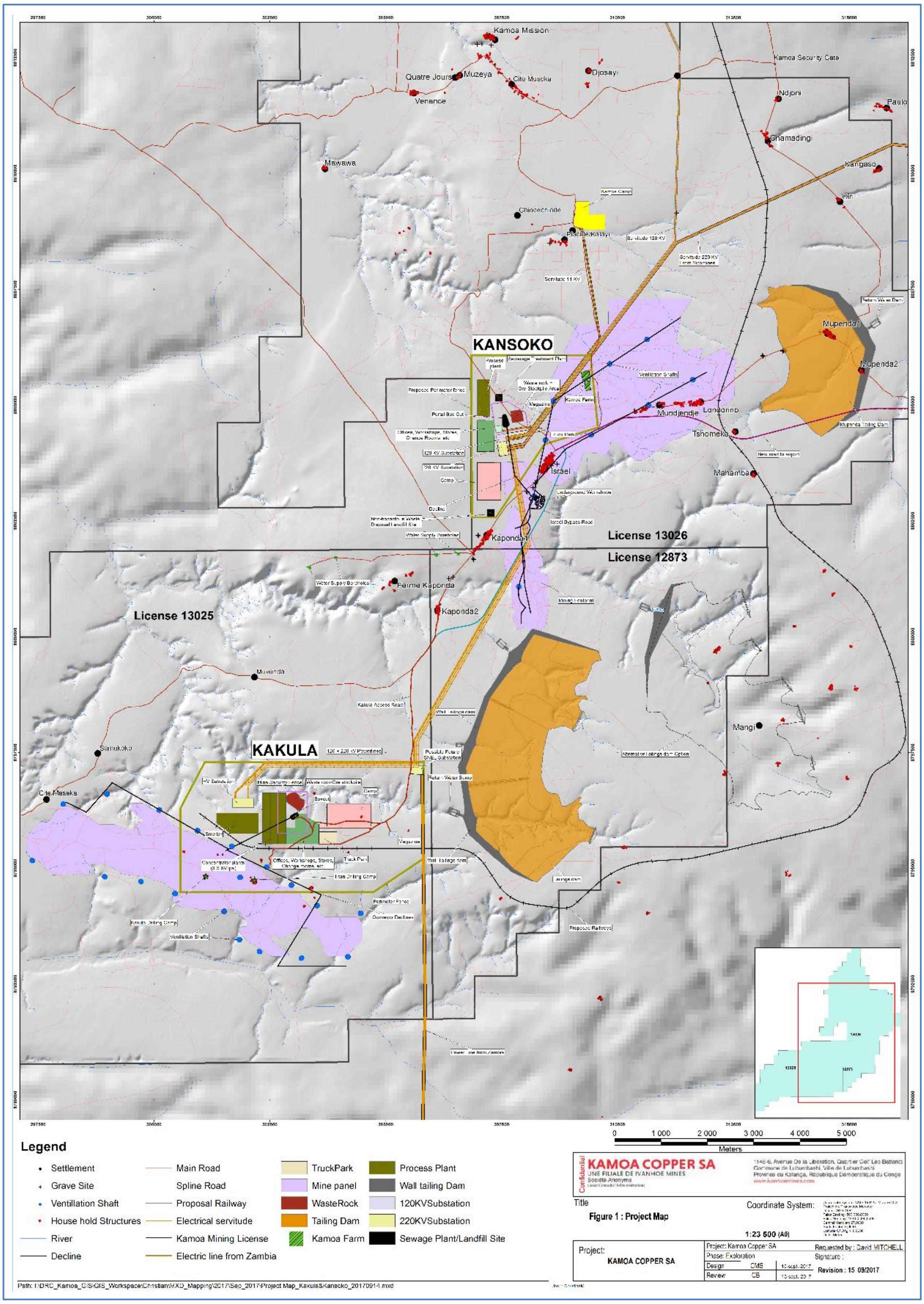

Figure by Kamoa Copper SA, 2017.

1.14.1 Kakula 6 Mtpa PEA Results Summary

The Kakula 6 Mtpa PEA represents the initial phase of the Kakula development. The Kakula 2017 PEA evaluates the development of a 6 Mtpa underground mine and surface processing complex at the Kakula Deposit – a discovery announced in early 2016 – as the project's first phase of development. The development scenario of the Kakula Mine on the Kakula Deposit is shown in Figure 1.9.

This PEA analyses the potential development of an initial 6 Mtpa Kakula Mine at the Kakula Deposit in the southerly portion of the Kamoa-Kakula Project's discovery area. For this option, the PEA envisages an average annual production rate of 284,000 tonnes of copper at a mine site cash cost of US$0.51/lb copper and total cash cost of US$1.14/lb copper for the first ten years of operations, and copper annual production of up to 320,000 tonnes by Year 9. The pre-production capital cost of US$1.2 billion for this option would result in an after-tax net present value at an 8% discount rate (NPV8%) of US$4.2 billion.

Figure 1.9 Kakula 2017 PEA 6 Mtpa Development Scenario

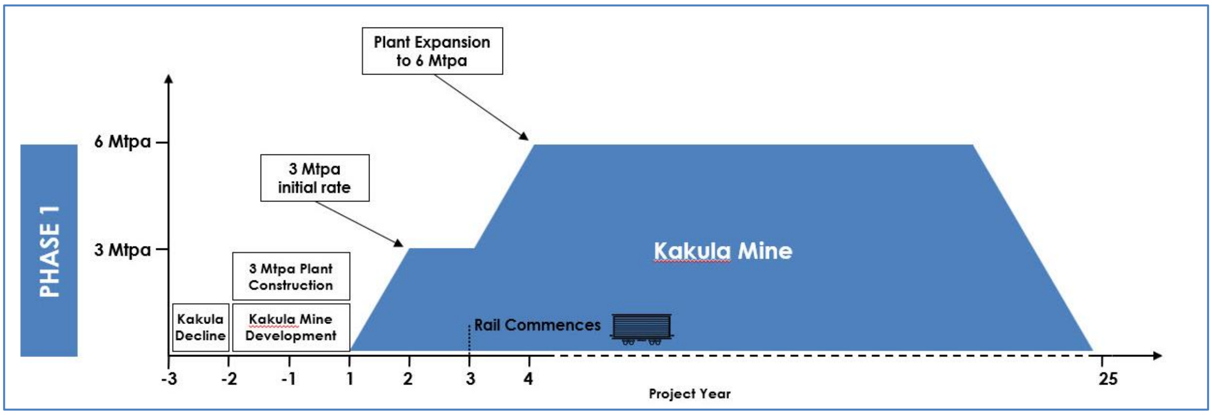

Figure by OreWin, 2017.

A summary of the key results for the Kakula 6 Mtpa PEA scenario are:

- Very-high-grade initial phase of production is projected to have a grade of 7.3% copper in Year Four and an average grade of 6.4% copper over the initial 10 years of operations, resulting in estimated average annual copper production of 284,000 tonnes.
- Annual copper production is estimated at 385,000 tonnes in Year Four.
- Initial capital cost, including contingency, is estimated at US$1.2 billion.
- Average total cash cost of US$1.14/lb of copper during the first 10 years.
- After-tax NPV, at an 8% discount rate, of US$4.2 billion.
- After-tax internal rate of return (IRR) of 36.2%, and a payback period of 3.1 years.
- Kakula is expected to produce a very-high-grade copper concentrate in excess of 50% copper, with extremely low arsenic levels.

The 6 Mtpa PEA assesses the potential development of the Kakula Deposit as a 6 Mtpa mining and processing complex. The Kakula mill would be constructed in two smaller phases of 3 Mtpa each as the mining operations ramp-up to full production of 6 Mtpa. The life-of-mine production scenario provides for 108.4 million tonnes to be mined at an average grade of 5.48% copper, producing 9.4 million tonnes of high-grade copper concentrate, containing approximately 11.4 billion pounds of copper.

The economic analysis uses a long-term price assumption of US$3.00/lb of copper and returns an after-tax NPV at an 8% discount rate of US$4.2 billion. It has an after-tax IRR of 36.2% and a payback period of 3.1 years.

The estimated initial capital cost, including contingency, is US$1.2 billion. The capital expenditure for off-site power, which is included in the initial capital cost, includes a US$71 million advance payment to the DRC state-owned electricity company, SNEL, to upgrade two hydropower plants (Koni and Mwadingusha) to provide the Kamoa-Kakula Project with access to clean electricity for its planned operations. Mwadingusha is being upgraded first. The work is being led by Stucky Ltd., of Switzerland; the advance payment will be recovered through a reduction in the power tariff.

The Kakula 2017 PEA is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves – and there is no certainty that the results will be realised. Mineral Resources do not have demonstrated economic viability and are not Mineral Reserves.

Key results of the Kakula 2017 PEA for a single 6 Mtpa mine are summarised in Table 1.9.

The Kamoa-Kakula 2018 Resource Update includes economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability. The results of the Kamoa-Kakula 2018 Resource Update represent forward looking information. The forward-looking information includes metal price assumptions, cash flow forecasts, projected capital and operating costs, metal recoveries, mine life and production rates, and other assumptions used in the Kamoa-Kakula 2018 Resource Update. Readers are cautioned that actual results may vary from those presented. The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially are presented in the body of this report under each relevant section.

Table 1.9 Kakula Mine Results Summary for 6 Mtpa Production

Item	Unit	Total
Total Processed		
Quantity Milled	kt	108,422
Copper Feed Grade	%	5.48
Total Concentrate Produced		
Copper Concentrate Produced	kt (dry)	9,400
Copper Recovery	%	86.86
Copper Concentrate Grade	%	54.94
Contained Metal in Concentrate	Mlb	11,385
Contained Metal in Concentrate	kt	5,164
Peak Annual Contained Metal in Concentrate	kt	385
10-Year Average		
Copper Concentrate Produced	kt (dry)	517
Contained Metal in Concentrate	kt	284
Mine-Site Cash Cost	US$/lb	0.51
Total Cash Cost	US$/lb	1.14
5-Year Average		
Copper Concentrate Produced	kt (dry)	448
Contained Metal in Concentrate	kt	246
Mine-Site Cash Cost	US$/lb	0.45
Total Cash Cost	US$/lb	1.08
Key Financial Results		
Peak Funding	US$M	1,135
Initial Capital Cost	US$M	1,231
Expansion Capital Cost	US$M	318
Sustaining Capital Cost	US$M	1,443
LOM Average Mine Site Cash Cost	US$/lb Cu	0.60
LOM Average Total Cash Cost	US$/lb Cu	1.23
Site Operating Cost	US$/t Milled	61.49
After-Tax NPV8%	US$M	4,243
After-Tax IRR	%	36.2
Project Payback Period	Years	3.1
Initial Project Life	Years	24

Table 1.10 summarizes the financial results. The mining production statistics are shown in Table 1.11. The Kakula 2017 PEA 6 Mtpa mill feed and copper grade profile for the first 20 years are shown in Figure 1.10 and the concentrate and metal production for the first 20 years are shown in Figure 1.11.

Table 1.10 Kakula 6 Mtpa PEA Financial Results

Net Present Value (US$M)	Discount Rate	Before Taxation	After Taxation
	Undiscounted	16,607	11,700
	4.0%	9,940	6,919
	6.0%	7,816	5,398
	8.0%	6,200	4,243
	10.0%	4,955	3,353
	12.0%	3,984	2,660
Internal Rate of Return	–	43.0%	36.2%
Project Payback Period (Years)	–	2.9	3.1

Table 1.11 Kakula 6 Mtpa PEA Production and Processing

Item	Unit	Years 1-5	Years 1-10	LOM Average
Total Processed				
Quantity Milled	kt	4,135	5,073	4,518
Copper Feed Grade	%	6.80	6.42	5.48
Annual Concentrate Produced				
Copper Concentrate Produced	kt (dry)	448	517	392
Copper Recovery	%	87.46	87.29	86.86
Copper Concentrate Grade	%	54.94	54.94	54.94
Annual Contained Metal in Concentrate				
Copper	Mlb	543	627	474
Copper	kt	246	284	215
Annual Payable Metal				
Copper	Mlb	530	612	463
Copper	kt	240	277	210

Figure 1.10 Kakula 6 Mtpa PEA Process Production

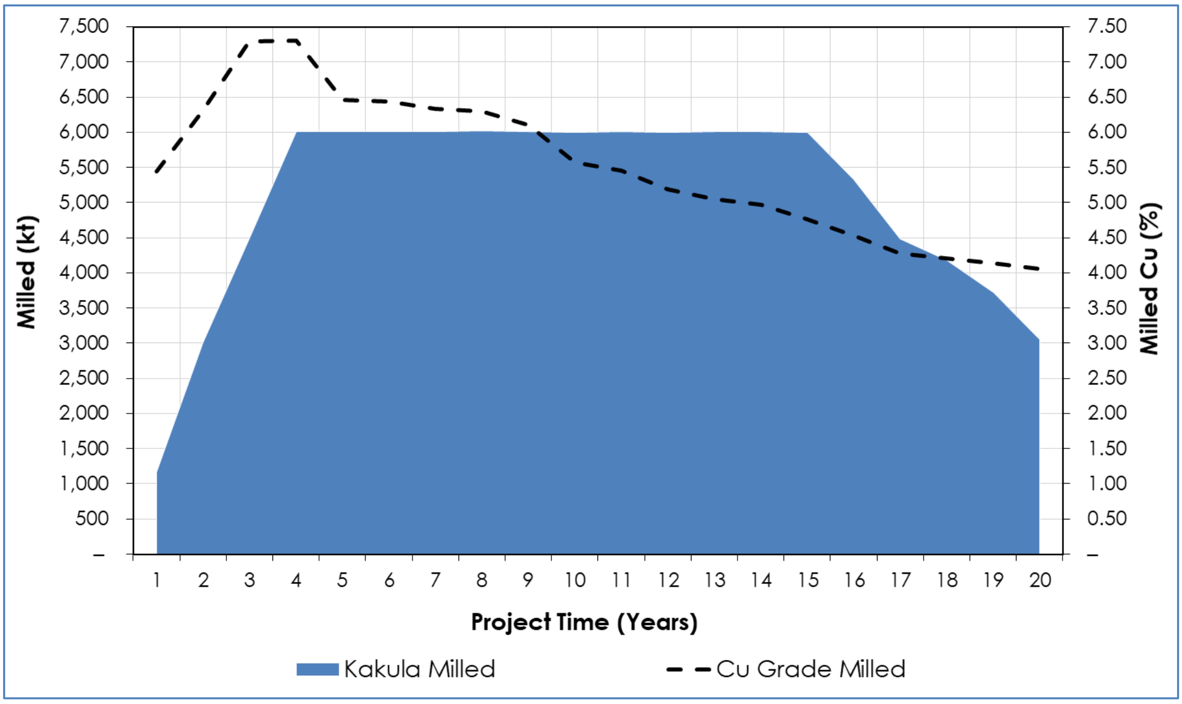

Figure by OreWin, 2017.

Figure 1.11 Kakula 6 Mtpa PEA Concentrate and Metal Production

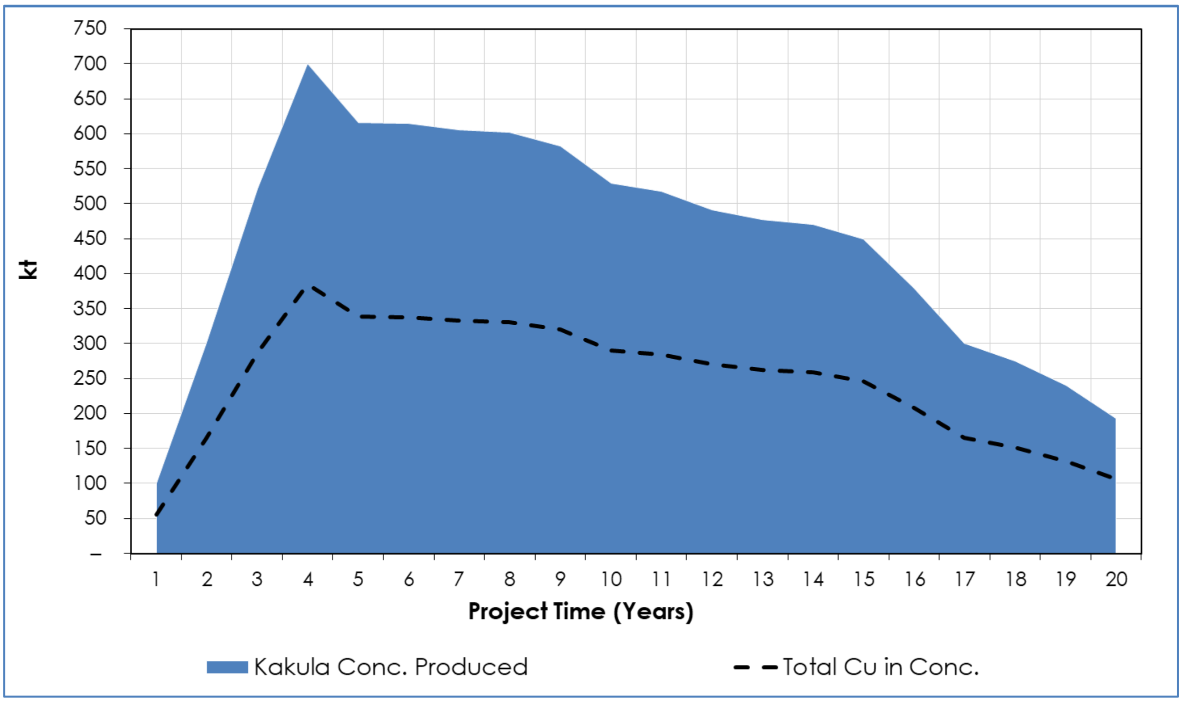

Figure by OreWin, 2017.

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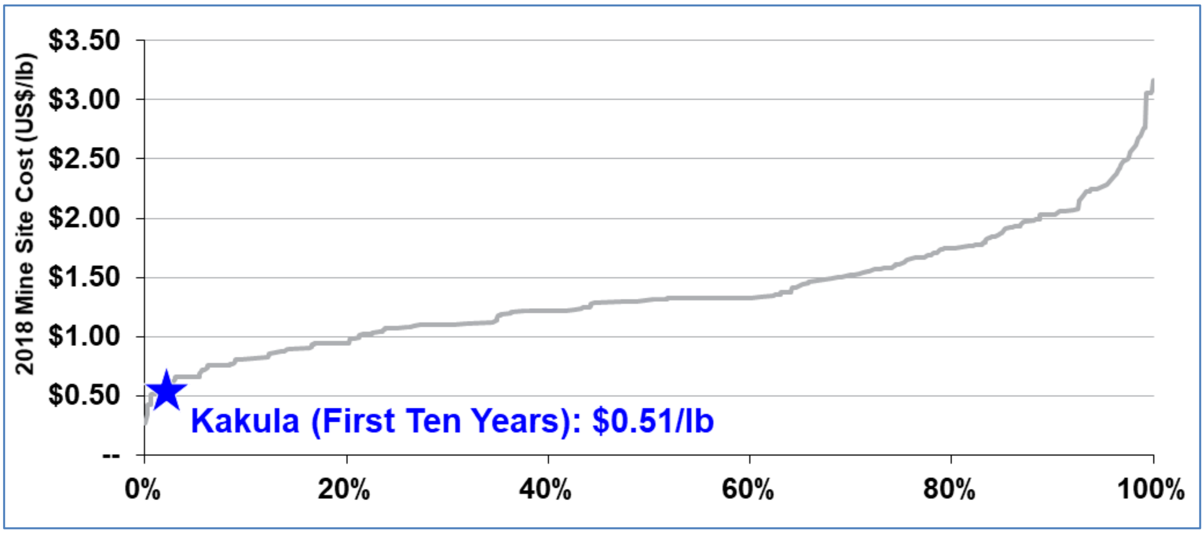
Table 1.12 summarizes unit operating costs. Figure 1.12 compares the average mine-site cash cost during the first 10 years of the Kakula 2017 PEA and Wood Mackenzie's comparable projects and Figure 1.14 compares the C1 pro-rata copper cash costs of the Kakula 2017 PEA and Wood Mackenzie's comparable projects.

Table 1.12 Kakula 6 Mtpa PEA Unit Operating Costs

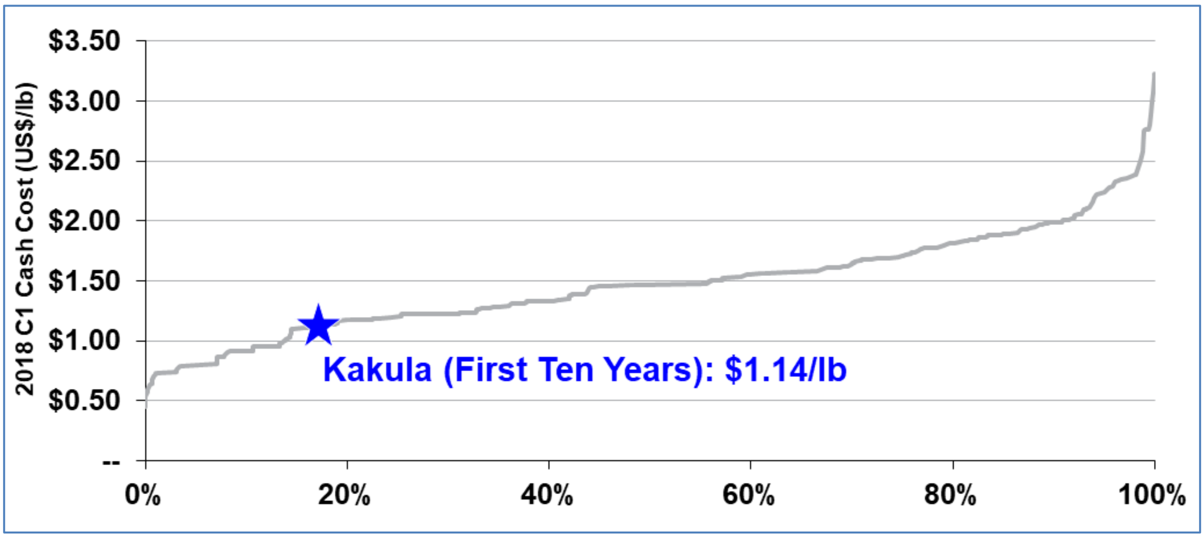
	US$/lb Payable Copper		
	Years 1-5	**Years 1-10**	**LOM Average**
Mine Site	0.45	0.51	0.60
Transport	0.31	0.31	0.31
Treatment and Refining Charges	0.15	0.15	0.15
Royalties and Export Tax	0.17	0.17	0.17
Total Cash Costs	**1.08**	**1.14**	**1.23**

Figure 1.12 2018 Mine-Site Cash Costs (Includes All Operational Costs at Mine Site)



Note: Represents mine-site cash costs that reflect the direct cash costs of producing paid concentrate or cathode incorporating mining, processing and mine-site G&A costs. Kakula is based on the average mine-site cash cost during the first 10 years as detailed in the Kakula 2017 PEA. Source: Wood Mackenzie (based on public disclosure, the Kakula 2017 PEA has not been reviewed by Wood Mackenzie).

Figure 1.13 2018 C1 Copper Cash Costs



Note: Represents C1 pro-rata cash costs that reflect the direct cash costs of producing paid metal incorporating mining, processing, mine-site G&A and offsite realization costs, having made appropriate allowance for the costs associated with the co-product revenue streams. Kakula is based on the average total cash cost during the first 10 years as detailed in the Kakula 2017 PEA. Source: Wood Mackenzie (based on public disclosure, the Kakula 2017 PEA has not been reviewed by Wood Mackenzie).

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

Table 1.13 Kakula 6 Mtpa PEA Revenue and Operating Costs

	Total LOM	Years 1-5	Years 1-10	LOM Average
	US$M	US$/t Milled		
Revenue				
Copper in Concentrate	33,346	384.31	361.76	307.56
Gross Sales Revenue	33,346	384.31	361.76	307.56
Less: Realization Costs				
Transport	3,418	39.93	37.21	31.52
Treatment and Refining	1,663	19.16	18.04	15.34
Royalties and Export Tax	1,935	22.29	20.99	17.85
Total Realization Costs	7,015	81.38	76.24	64.70
Net Sales Revenue	26,331	302.93	285.53	242.86
Site Operating Costs				
Underground Mining	4,679	39.94	44.65	43.16
Processing	1,308	12.00	12.14	12.06
Tailings	29	0.30	0.25	0.26
General and Administration	728	6.36	5.77	6.71
SNEL Discount	-187	-2.12	-2.23	-1.67
Customs	104	0.91	0.99	0.96
Total	6,661	57.38	61.57	61.49
Net Operating Margin	19,670	245.55	223.96	181.37
Net Operating Margin	74.70%	81.06%	78.44%	74.68%

Table 1.14 Kakula 6 Mtpa PEA Capital Costs

Description	Initial Capital	Expansion Capital	Sustaining Capital	Total
	US$M	US$M	US$M	US$M
Mining				
Underground Mining	403	–	1,045	1,447
Capitalised Pre-Production	36	–	–	36
Subtotal	438	–	1,045	1,483
Power				
Power Supply Off Site	71	–	–	71
Capitalised Power Cost	4	–	–	4
Subtotal	75	–	–	75
Concentrate and Tailings				
Process Plant	146	84	159	389
Tailings	27	74	–	101
Subtotal	173	158	159	489
Infrastructure				
Mine Surface Infrastructure	35	–	24	59
General Infrastructure	110	–	76	187
Rail Link	–	48	–	48
Subtotal	145	48	100	293
Indirects				
EPCM	78	31	–	109
Owners Cost	95	20	–	115
Closure	–	–	75	75
Subtotal	173	51	75	298
Capital Expenditure Before Contingency	1,004	257	1,378	2,638
Contingency	227	62	65	354
Capital Expenditure After Contingency	1,231	318	1,443	2,992

Figure 1.14 compares the capital intensity for Large-Scale Copper Projects of Wood Mackenzie's projects currently in construction. The figure shows recently approved projects and other projects rated in the Wood Mackenzie database to be developed with nominal copper production capacity in excess of 200 ktpa. The estimates are based on public disclosure and information gathered by Wood Mackenzie. The Kamoa-Kakula 2017 Development Plan was not reviewed by Wood Mackenzie prior to filing.

Figure 1.14 Capital Intensity for Large-Scale Copper Projects

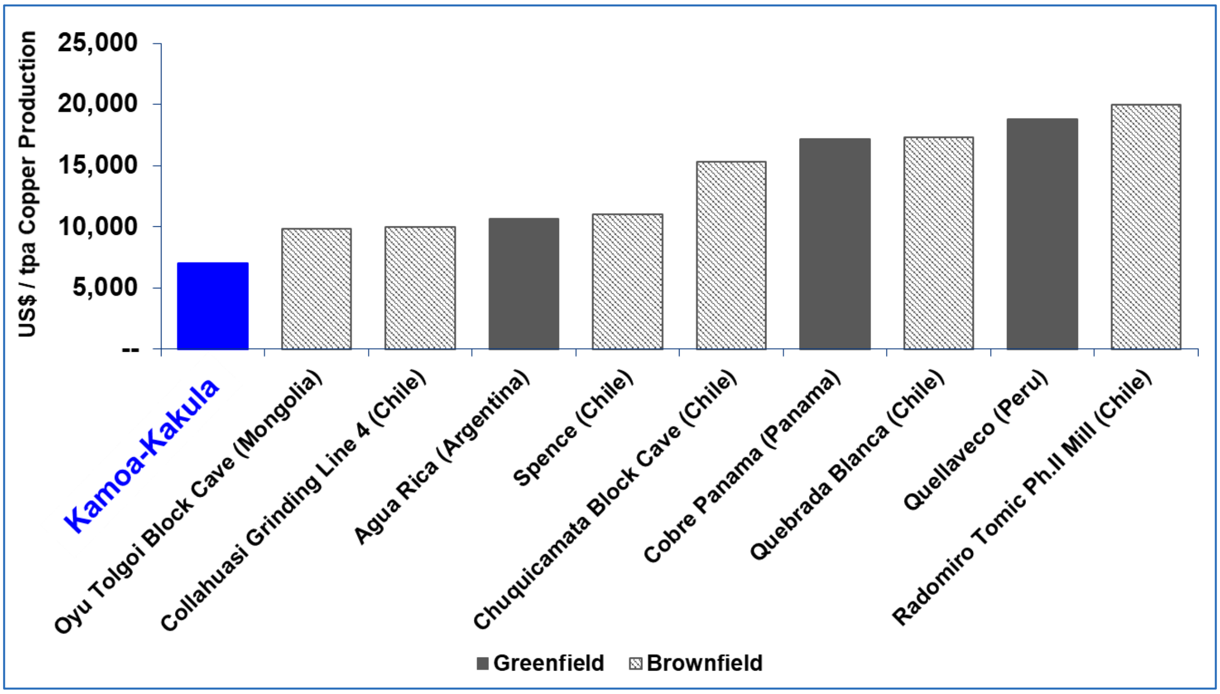


Figure by Ivanhoe, 2017. Source: Wood Mackenzie.

The after-tax NPV sensitivity to metal price variation is shown in Table 1.15 for copper prices from US$2.00/lb to US$4.00/lb. The annual and cumulative cash flows are shown in Figure 1.15 (annual cash flow is shown on the left vertical axis and cumulative cash flow on the right axis).

Table 1.15 Kakula Mine Copper Price Sensitivity

After-Tax NPV (US$M)	Copper Price - US$/lb				
Discount Rate	**2.00**	**2.50**	**3.00**	**3.50**	**4.00**
Undiscounted	4,135	7,921	11,700	15,478	19,253
4.0%	2,257	4,591	6,919	9,247	11,573
6.0%	1,654	3,529	5,398	7,267	9,135
8.0%	1,195	2,722	4,243	5,764	7,282
10.0%	841	2,100	3,353	4,606	5,856
12.0%	567	1,617	2,660	3,703	4,744
IRR	18.9%	28.6%	36.2%	42.8%	48.6%

Figure 1.15 Kakula Mine Projected Cumulative Cash Flow

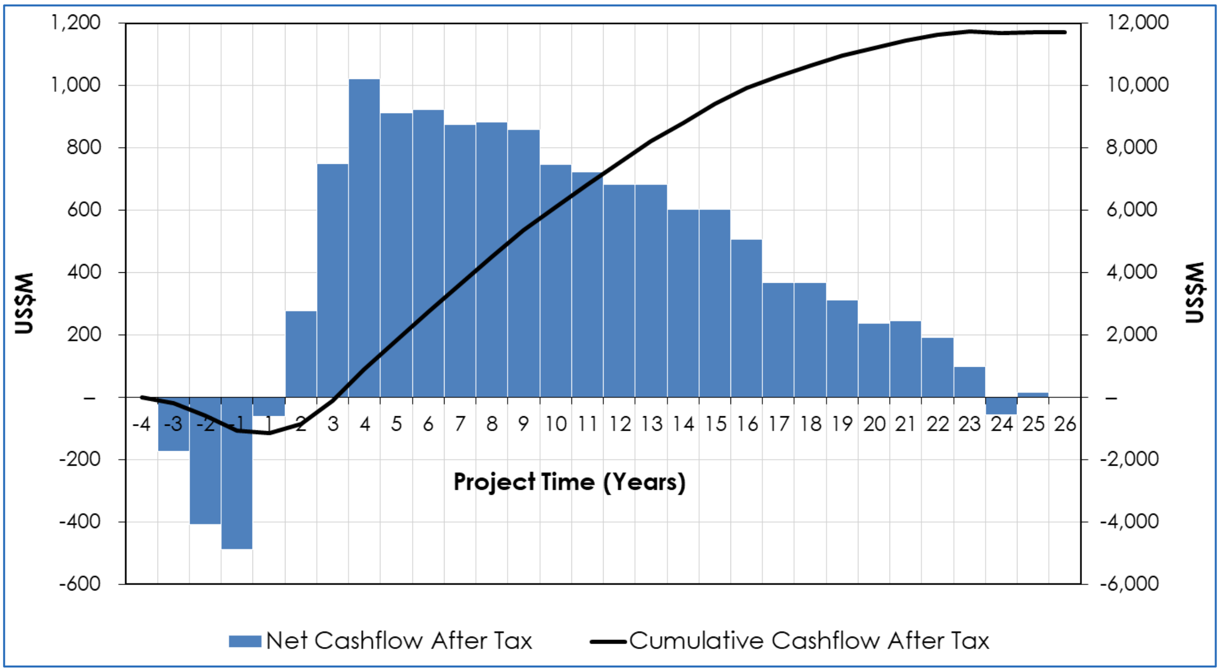


Figure by OreWin, 2017.

1.14.2 Kamoa-Kakula 12 Mtpa PEA Results Summary

The Kamoa-Kakula 12 Mtpa PEA scenario assesses the development of both the Kakula and Kamoa deposits as an integrated, 12 Mtpa mining and processing complex. Each operation is expected to be a separate underground mine with associated dedicated processing facilities and surface infrastructure. The Kamoa-Kakula 2017 PEA 12 Mtpa scenario envisages the construction and operation of two separate facilities: the Kakula Mine on the Kakula Deposit and the Kansoko Mine on the Kansoko Sud and Kansoko Centrale areas of the Kamoa Deposit. The Kakula Mine scenario is the same as that presented in the Kakula 2017 PEA 6 Mtpa. The initial plant capacity of 3 Mtpa is expanded to 6 Mtpa as the Kansoko Mine and Kakula Mine ramp up. The mines continue to ramp up to 12 Mtpa combined by Year Nine. Once the Kansoko and Kakula Mines near the end of their mine life, Kamoa North comes on line to maintain the overall production at 12 Mtpa. The 12 Mtpa PEA also analyses an on-site smelter to produce blister copper at the mine site.

The Kamoa-Kakula 12 Mtpa PEA has US$1.2 billion in initial capital costs. Future expansion at the Kansoko Mine and subsequent extensions could be funded by cash flows from the Kakula Mine, resulting in an after-tax net present value at an 8% discount rate (NPV8%) of US$7.2 billion and an internal rate of return of 33%. Under this approach, the 12 Mtpa PEA also includes the construction of a direct-to-blister flash copper smelter with a capacity of 690,000 tonnes of copper concentrate per annum to be funded from internal cash flows. This would be completed in Year Five of operations, achieving significant savings in treatment charges and transportation costs.

The Kamoa-Kakula 12 Mtpa PEA scenario has average annual production of 370,000 tonnes of copper at a total cash cost of US$1.02/lb copper during the first 10 years of operations and production of 542,000 tonnes by Year Nine. At this future production rate, Kamoa-Kakula would rank among the world's five largest copper mines.

A summary of the key results for the Kamoa-Kakula 12 Mtpa PEA scenario are:

- Very-high-grade initial phase projected to have a grade of 7.3% copper in Year Four and an average grade of 5.72% copper during the first 10 years of operations, resulting in estimated average annual copper production of 370,000 tonnes.
- Annual copper production is estimated at 542,000 tonnes in Year Nine, ranking Kamoa-Kakula as potentially one of the five largest copper mines in the world.
- Initial capital cost, including contingency, is US$1.2 billion, with subsequent expansions from Kansoko and other mining areas, as well as the smelter, to be funded by cash flows from the Kakula Mine.
- Average total cash costs of US$1.02/lb of copper during the first 10 years, including sulphuric acid credits.
- After-tax NPV, at an 8% discount rate, of US$7.2 billion.
- After-tax IRR of 33% and a payback period of 4.7 years.

The Kamoa-Kakula 12 Mtpa PEA development scenario and long-term development plan is shown in Figure 1.16. Key results of the Kamoa-Kakula 12 Mtpa PEA Scenario are summarised in Table 1.16. The production results for the external smelter (concentrate sales to off-site customers) and internal smelter (on-site smelter owned by the project) scenarios are both shown in Table 1.17.

The Kamoa-Kakula 2018 Resource Update includes economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability. The results of the Kamoa-Kakula 2018 Resource Update represent forward looking information. The forward-looking information includes metal price assumptions, cash flow forecasts, projected capital and operating costs, metal recoveries, mine life and production rates, and other assumptions used in the Kamoa-Kakula 2018 Resource Update. Readers are cautioned that actual results may vary from those presented. The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially are presented in the body of this report under each relevant section.

Figure 1.16 Kamoa-Kakula 2017 PEA Long-Term Development Plan

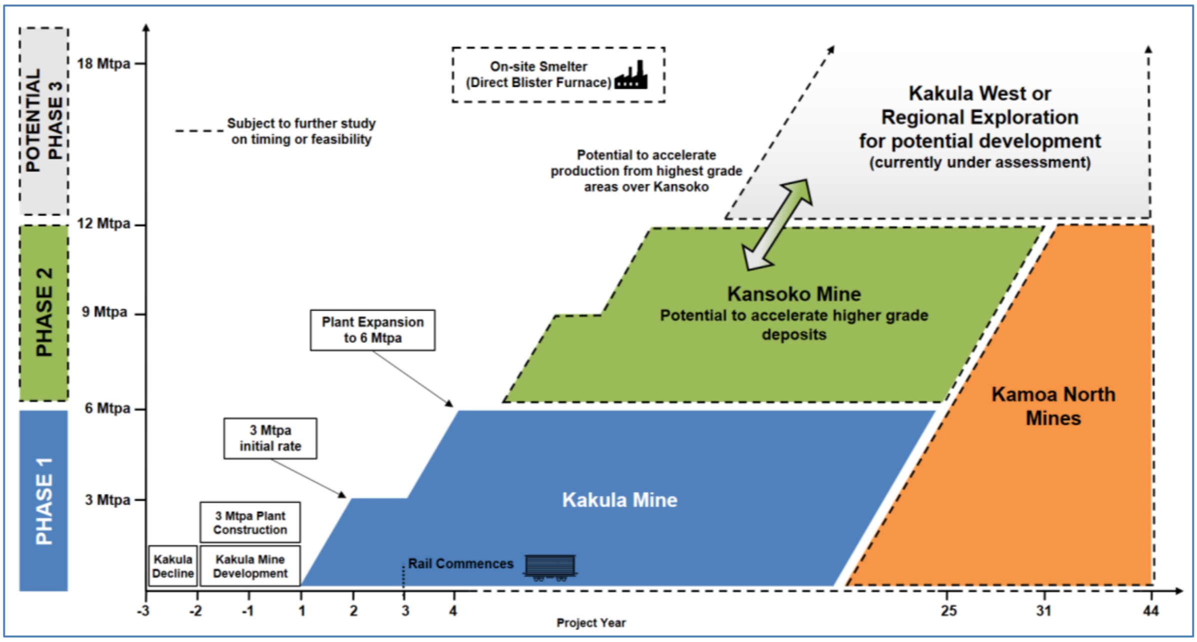


Figure by OreWin, 2017.

Table 1.16 Results Summary – Kamoa-Kakula 12 Mtpa PEA

Item	Unit	Total
Total Processed		
Quantity Milled	kt	444,276
Copper Feed Grade	%	3.79
Copper Concentrate Produced	kt (dry)	34,206
Copper Concentrate - External Smelter	kt (dry)	9,744
Copper Concentrate - Internal Smelter	kt (dry)	24,461
Copper Recovery	%	85.97
Copper Concentrate Grade	%	42.30
Cont. Metal in Conc. - External Smelter	Mlb	10,627
Cont. Metal in Conc. - External Smelter	kt	4,820
Cont. Metal in Blister - Internal Smelter	Mlb	20,955
Cont. Metal in Blister - Internal Smelter	kt	9,505
Peak Annual Contained Metal in Concentrate	kt	542
10 Year Average		
Copper Feed Grade	%	5.72
Copper Concentrate Produced	kt (dry)	758
Cont. Metal in Conc. - External Smelter	kt	188
Cont. Metal in Blister - Internal Smelter	kt	182
Mine Site Cash Cost (Including Smelter)	US$/lb	0.63
Total Cash Cost (After Credits)	US$/lb	1.02
Key Financial Results		
Peak Funding	US$M	1,139
Initial Capital Cost	US$M	1,235
Expansion Capital Cost	US$M	3,647
Sustaining Capital Cost	US$M	5,133
LOM Avg. Mine Site Cash Cost (Including Smelter)	US$/lb	0.91
LOM Avg. Total Cash Costs (After Credits)	US$/lb	1.20
Site Operating Cost	US$/t Milled	64.17
After-Tax NPV8%	US$M	7,179
After-Tax IRR	%	33.0
Project Payback	Years	4.7
Initial Project Life	Years	44

Table 1.17 Kakula 12 Mtpa PEA Production and Processing

Item	Unit	TOTAL LOM	YEARS 1-5	YEARS 1-10	LOM AV.
Total Processed					
Quantity Milled	kt	444,276	4,369	7,442	10,097
Copper Feed Grade	%	3.79	6.63	5.72	3.79
Concentrate Produced					
Copper Concentrate Produced	kt (dry)	34,206	467	758	777
Copper Concentrate - External Smelter	kt (dry)	9,744	329	344	221
Copper Concentrate - Internal Smelter	kt (dry)	24,461	138	414	556
Copper Recovery	%	85.97	87.47	87.51	85.97
Copper Concentrate Grade	%	42.30	54.17	49.18	42.30
Contained Metal in Concentrate - External Smelter					
Copper	Mlb	10,627	399	415	242
Copper	kt	4,820	181	188	110
Payable Metal in Concentrate - External Smelter					
Copper	Mlb	10,348	390	405	235
Copper	kt	4,694	177	184	107
Contained Metal in Blister - Internal Smelter					
Copper	Mlb	20,955	157	400	476
Copper	kt	9,505	71	182	216
Payable Metal in Blister - Internal Smelter					
Copper	Mlb	20,892	156	399	475
Copper	kt	9,476	71	181	215
Payable Metal					
Copper	Mlb	31,240	546	804	710
Copper	kt	14,170	248	365	322

Table 1.18 summarizes unit operating costs. The after-tax NPV sensitivity to metal price variation is shown in Table 1.19 for copper prices from US$2.00/lb to US$4.00/lb.

Table 1.18 Unit Operating Costs for Kamoa-Kakula 12 Mtpa PEA

	US$/lb Payable Copper		
	Years 1-5	Years 1-10	LOM Average
Mine Site (ex-Smelter)	0.46	0.54	0.78
Smelter	0.05	0.09	0.13
Transport	0.27	0.23	0.21
Treatment and Refining Charges	0.12	0.10	0.09
Royalties and Export Tax	0.15	0.13	0.12
Total Cash Costs Before Credits	1.04	1.09	1.33
Sulphuric Acid Credits[1]	(0.03)	(0.07)	(0.13)
Total Cash Costs After Credits	1.02	1.02	1.20

[1] Assumes a sulphuric acid price of US$200 per tonne.

Table 1.19 Copper Price Sensitivity for Kamoa-Kakula 12 Mtpa PEA

After-Tax NPV (US$M)	Copper Price (US$/lb)				
Discount Rate	**2.00**	**2.50**	**3.00**	**3.50**	**4.00**
Undiscounted	10,638	21,313	31,970	42,598	53,213
4.0%	4,540	9,414	14,283	19,146	24,005
6.0%	2,969	6,492	10,008	13,522	17,033
8.0%	1,913	4,549	7,179	9,808	12,435
10.0%	1,187	3,218	5,243	7,267	9,290
12.0%	679	2,282	3,879	5,475	7,069
IRR	16.6%	25.5%	33.0%	39.6%	45.5%

The Kamoa-Kakula 12 Mtpa PEA mill feed and copper grade profile are shown in Figure 1.17 and the concentrate and metal production are shown in Figure 1.18.

Figure 1.17 12 Mtpa PEA Scenario Mill Feed and Grade Profile

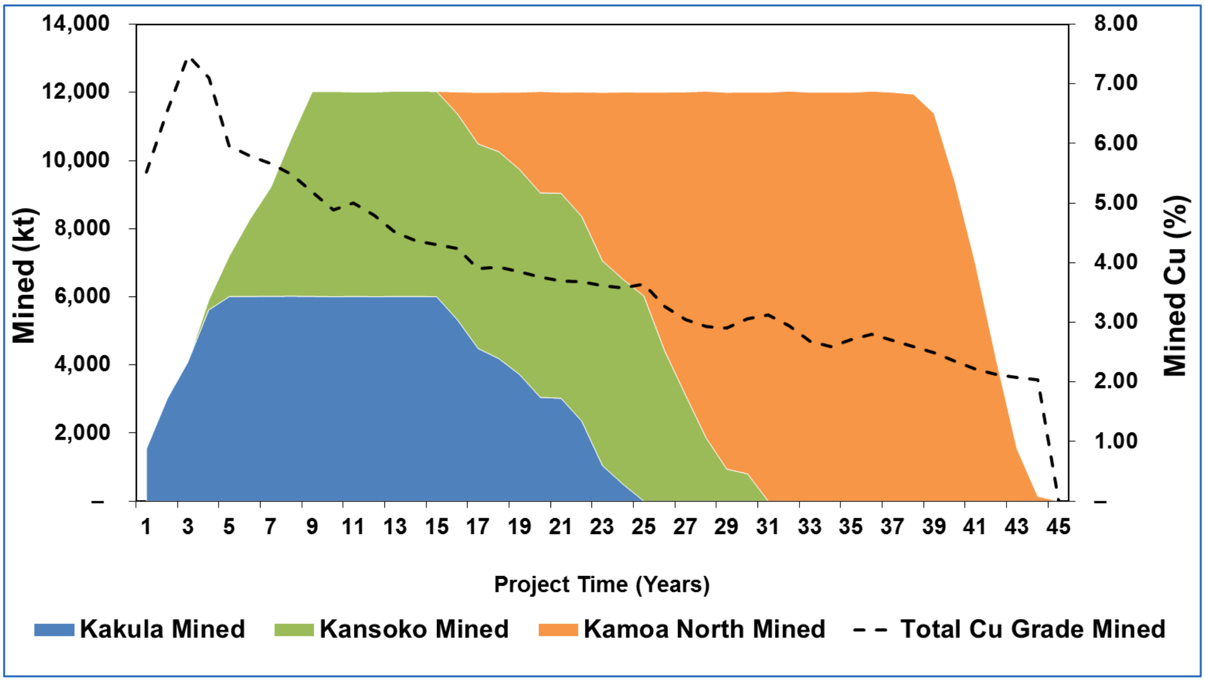


Figure by OreWin, 2017.

Figure 1.18 Kamoa-Kakula 12 Mtpa PEA Concentrate and Metal Production

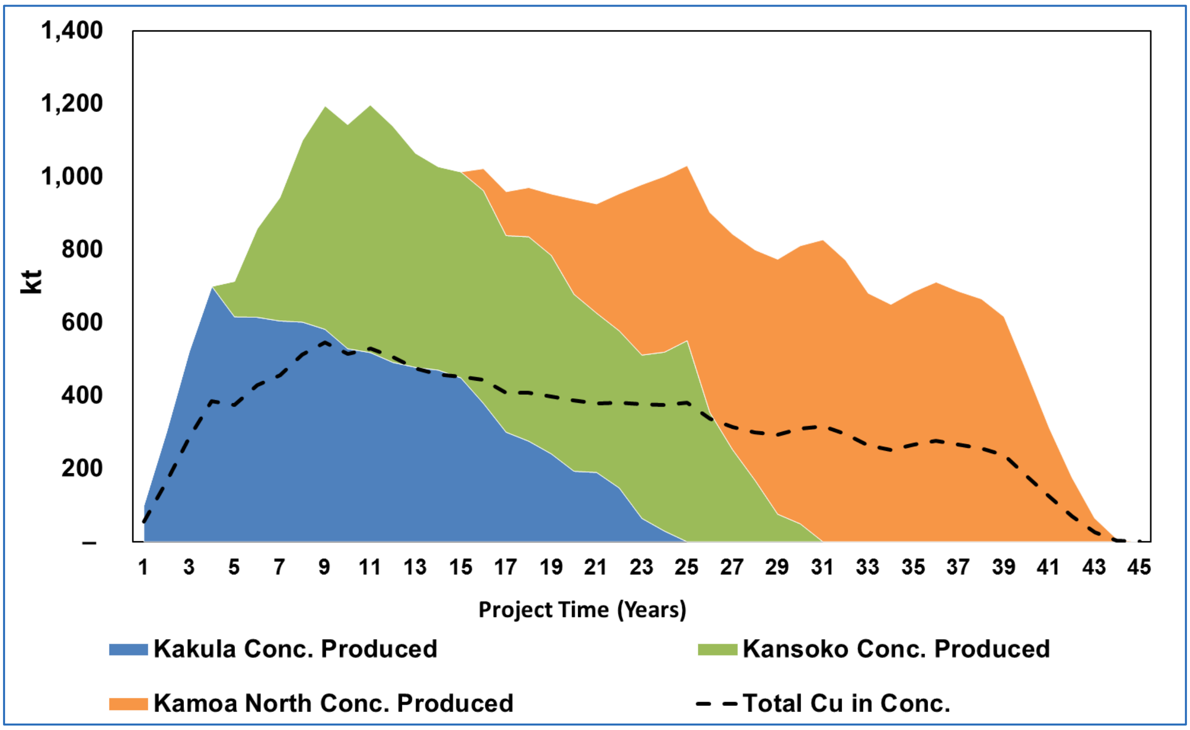


Figure by OreWin, 2017.

1.14.3 Kakula 6 Mtpa PEA Mining

Mining methods in the Kakula 2017 PEA are assumed to be a combination of the controlled convergence room-and-pillar mining method and drift-and-fill with pastefill mining method.

Selection of the mining method was dictated by mining height and dip. The controlled convergence room-and-pillar method was selected for heights greater than 3 m and less than 6 m, and dip less than 25 degrees. The drift-and-fill with pastefill was selected for heights greater than 6 m. The drift-and-fill with pastefill method was also selected for heights greater than 3 m and less than 6 m, and dip greater than 25 degrees.

Two drift-and-fill mining lifts were selected for the stope heights greater than 6.00 m. The maximum height of the drift-and-fill mining panel is 6 m, and the minimum height of the drift-and-fill mining panel is 3.00 m. Mine panels were also spilt by grade into primary and secondary mining zones for scheduling. Primary zones contain Cu>6.6% and secondary zones contain Cu<6.6% and Cu>3%. Figure 1.19 shows the location of the primary and secondary mining zones and the Kakula 2017 PEA 6 Mtpa development.

Figure 1.19 Kakula 2017 PEA Development and Mining Zones

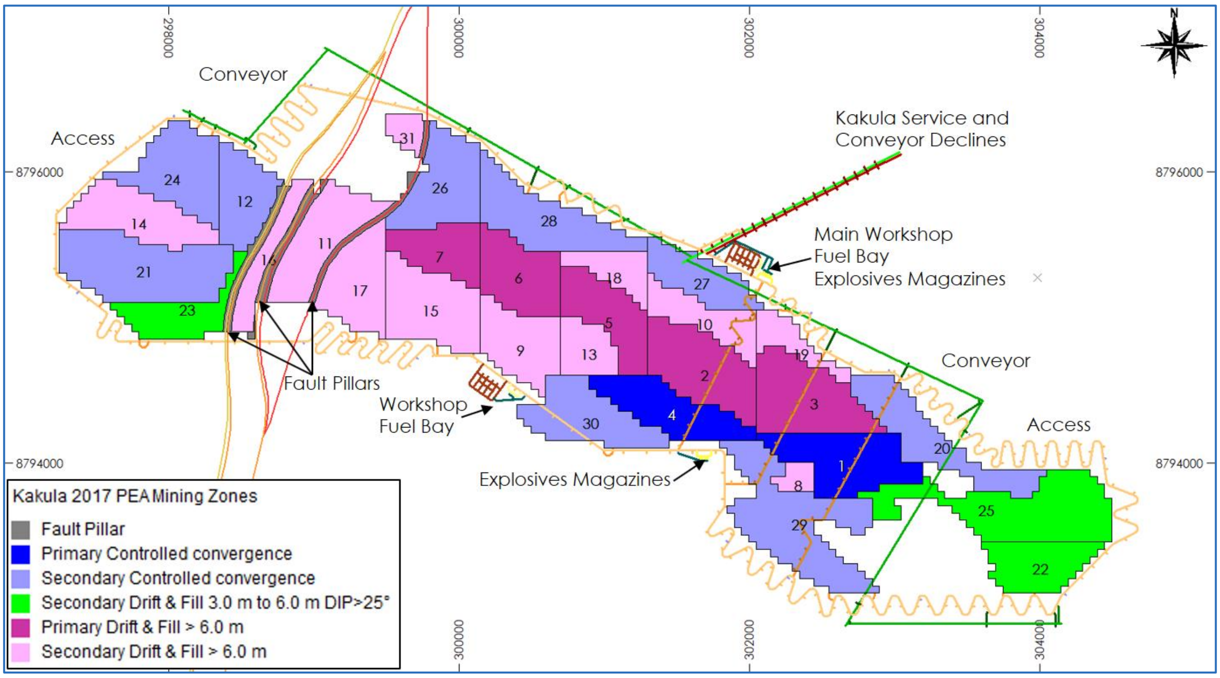


Figure by OreWin, 2017.

The Kakula 2017 PEA is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves – and there is no certainty that the results will be realised. Mineral Resources do not have demonstrated economic viability and are not Mineral Reserves.

1.14.4 Kakula Metallurgical Testwork and Concentrator Assumptions

The Kakula 6 Mtpa PEA requires a single 6 Mtpa processing plant based on only treating Kakula feed. The Kamoa-Kakula 12 Mtpa PEA requires the construction of two 6 Mtpa processing plants, one treating only Kakula feed and the other treating Kansoko ore.

The 6 Mtpa plant required to process Kansoko ore has been previously described in the Kamoa 2017 PFS and is summarised above in Section 1.13.2. The 6 Mtpa plant required to process Kakula feed has an identical flowsheet to the Kansoko plant with the non-material difference that the crushing circuit produces a -8 mm grinding circuit feed, rather than the -10 mm product generated with Kansoko ore.

Although the flowsheets will effectively be identical, the Kakula plant will require larger equipment capacities in the concentrate cleaning and concentrate handling parts of the plant, as the feed will be much higher-grade than at Kansoko. Some of the expansive effects of grade at Kakula are offset by the higher concentrate grade achieved. There is less Kakula concentrate mass to be treated per tonne of copper metal.

In the opinion of the Process QP, the metallurgical testwork performed to date on Kamoa, Kansoko and Kakula mineralisation samples is sufficient to support the PEA results. Although there are some outstanding technical issues that require better definition through testwork ahead of the production phase, none of these issues are material in the context of the project evaluations.

1.14.5 Power Supply

Electrical power for the Kamoa-Kakula Project is planned to be sourced on a priority basis from the DRC national grid in return for the financing of the rehabilitation of three hydropower plants: Koni, Mwadingusha, and Nzilo. A financing agreement with SNEL has been finalised for upgrading these plants to secure a long-term, clean, sustainable power supply to meet the requirements of the Kamoa-Kakula Project.

The Kakula 2017 PEA's estimated initial capital cost of US$1.0 billion includes a US$71 million (direct plus indirect costs) advance payment to SNEL to upgrade two of the hydropower plants, Koni and Mwadingusha, to provide the Kamoa-Kakula Project with hydroelectric power for its operations. The upgrading work is being led by Stucky Ltd., and the advance payment will be recovered through a reduction in the power tariff. The Kamoa-Kakula Project initially will be powered by existing capacity on the national grid, until upgrading work on the hydropower plants has been completed.

Three of six generators at the Mwadingusha hydropower plant have been upgraded as of January 2018, and the plant is supplying 32 MW of electricity to the national interconnected grid. The Kamoa-Kakula Project began drawing power from the national grid in October 2016.

1.15 Interpretation and Conclusions

1.15.1 Kamoa-Kakula 2018 Resource Update

The Kamoa-Kakula 2018 Resource Update provides an update of the Kamoa-Kakula Project Mineral Resource, with the Mineral Reserve from the Kamoa-Kakula 2017 Development Plan and the results of the preliminary economic assessment (PEA) from the Kamoa-Kakula 2017 PEA remaining the same. Aside from the updated Mineral Resource, further study work is currently incomplete and has not determined any results that require material changes to the Kamoa 2017 PFS or Kamoa-Kakula 2017 PEA.

Now that a Mineral Resource estimate has been independently verified for the Kakula West Discovery, Ivanhoe and Zijin can explore options to accelerate or expand future mine production by bringing high-grade mineralization from Kakula West into the Kakula mine plan.

Additional exploration success could have a significant influence on the size, value and timing of the overall development plan; as such, the Kamoa-Kakula development plans will be reassessed and amended as the project moves forward to reflect ongoing exploration results.

1.15.2 Mineral Resource Estimate

Mineral Resources for the Project have been estimated using core drill data, have been performed using industry best practices (CIM, 2003), and conform to the requirements of CIM Definition Standards (2014). Amec Foster Wheeler has checked the data used to construct the resource models, the methodology used to construct it (Datamine macros), and has validated the resource models. Amec Foster Wheeler finds the Kamoa and Kakula resource models to be suitable to support prefeasibility level mine planning.

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

- Drill spacing.
  - The drill spacing at the Kamoa and Kakula deposits is insufficient to determine the effects of local faulting on lithology and grade continuity assumptions. Local faulting could disrupt the productivity of a highly-mechanised operation. In addition, the amount of contact dilution related to local undulations in the SMZ has yet to be determined for both deposits. Ivanhoe plans to study these risks with the declines currently in progress at Kamoa. A similar decline is being developed to provide access to the Kakula deposit.
  - Delineation drill programs at the Kamoa deposit will have to use a tight (approximately 50 m) spacing to define the boundaries of mosaic pieces (areas of similar stratigraphic position of SMZs) in order that mine planning can identify and deal with these discontinuities. At the Kakula deposit, the mineralisation appears more continuous compared to Kamoa.
- Assumptions used to generate the data for consideration of reasonable prospects of eventual economic extraction for the Kamoa deposit.
  - Mining recovery could be lower and dilution increased where the dip locally increases on the flanks of the domes. The exploration decline should provide an appropriate trial of the conceptual room-and-pillar mining method on the Kamoa deposit in terms of costs, dilution, and mining recovery. The decline will also provide access to data and metallurgical samples at a bulk scale that cannot be collected at the scale of a drill sample.
- Assumptions used to generate the data for consideration of reasonable prospects of eventual economic extraction for the Kakula deposit.
  - A controlled convergence room-and-pillar technique is being studied which provides the opportunity for reduced costs.
- Metallurgical recovery assumptions at Kamoa.
  - Metallurgical testwork at the Kamoa deposit indicates the need for multiple grinding and flotation steps. Variability testwork has been conducted on only portions of the Kamoa deposit. Additional variability testing is needed to build models relating copper mineralogy to concentrate grade and improve the recovery modelling.
  - A basic model predicting copper recovery from certain supergene mineralisation types has been developed. More variability testing is required to improve this model to the point where it is useful for production planning purposes.
- Metallurgical recovery assumptions at Kakula.
  - Preliminary metallurgical testwork at the Kakula deposit indicates that a high-grade chalcocite-dominant concentrate could be produced at similar or higher recoveries compared to those achieved for Kamoa samples.
  - There is no supergene mineralisation currently identified at Kakula that requires a dedicated recovery model separate from the hypogene recovery prediction method.
- Exploitation of the Kamoa-Kakula Project requires building a greenfields project with attendant infrastructure. Changes in the assumptions as to operating and capital costs associated with the proposed development may affect the base case cut-off grades selected for the Kamoa and Kakula Mineral Resource estimates.

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

1.15.3 Kamoa-Kakula Development Plan

The development of Kamoa-Kakula should be reassessed for the impact of the Kakula West Mineral Resource in order to determine the relative value of Kakula West against the other areas within the Kakula and Kamoa Mineral Resources.

The Kamoa-Kakula 2017 Development Plan included an update of the Kamoa Mineral Reserve and updates of the preliminary economic assessment (PEA) on the Kakula Mineral Resource. The production rate assumption at each deposit has increased from 4 Mtpa to 6 Mtpa, and the total combined production rate has increased from 8 Mtpa to 12 Mtpa.

The Mineral Reserves for the Kamoa 2017 PFS increased as a result of an increase in production rate through a change to the controlled convergence room-and-pillar mining method.

The analysis in the Kakula 2017 PEA indicates that discovery of the Kakula deposit has changed the potential development scenarios for the Kamoa-Kakula project, and additional studies should be prepared to define the development sequence and production rates including mining methods, plant sizing and location for the deposits.

The Kamoa-Kakula 2018 Resource Update is an update of the Kakula Mineral Resource including Kakula West. The development scenario should be tested and reviewed to determine updates to the development plan.

1.15.4 Mineral Reserve Estimation

Mineral Reserves for the Kamoa 2017 PFS conform to the requirements of CIM Definition Standards (2014).

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

- The testing of the controlled convergence room-and-pillar mining method to the Kamoa deposit.
- Any changes to the resource model as a result of further definition drilling at the site.
- The availability of reliable power to the site.

1.16 Recommendations

1.16.1 Further Assessment

Ivanhoe now has three areas within the Kamoa-Kakula Project (Kamoa, Kakula and Kakula West) that warrant further assessment and are at different stages of study and development. Kakula is a very high-grade Mineral Resource that is separate to Kamoa and could be developed as a separate mine and processing facility, and given this, further study should be undertaken. The Kakula 2017 PEA has identified potential development scenarios for Kamoa and Kakula deposits.

The findings and recommendations of the Kamoa 2017 PFS remain current, and further studies on the Kamoa deposit are in progress but are not yet complete.

The recommendations from the Kamoa 2017 PFS were that a whole of project approach should be undertaken to optimise the project and to take the project through the study phases to production. The next phase of study should be to prepare a PFS on Kakula. These additional studies will assist in further defining the scope for the next studies of the overall development of the entire Kamoa-Kakula Project. The key areas for further studies are:

- Prepare the PFS of the Kakula deposit based on the updated Kakula Mineral Resource.
- Analyse and determine the Kakula West Mineral Resource and determine if a PEA is required.
- Revisions and updates of the long-term whole of project planning as the Mineral Resources are further defined. Including expanding and optimising the project production rate by considering concentrator and smelter capacities that are matched to the power supply availability, mine production and transport options.
- Other mining areas and additional mines from the Kamoa deposit.
- Rail transport to Lobito.
- Continue infill drilling programme to upgrade resource categorisation, to enhance the geotechnical database and its application to mine design and ground support, and to better understand the continuity of the deposit and impacts on productivities and dilution.
- Consider an underground exploration programme at Kamoa to attain first-hand information on actual mining conditions and to validate design assumptions.
- Complete hydrological studies and data evaluation to better determine impacts on underground mining conditions and productivities.

1.16.2 Drill Programme

An initial drill programme that was planned to consist of 129,000 m at a cost of US$21.2 M that commenced in May 2016 at Kakula. Up to the end of 2017, 177,860 m had been completed by both in-house drilling and contractor rigs.

Drilling is planned to continue at a similar rate in 2018. Amec Foster Wheeler has recommended a total programme of approximately 109,000 metres planned at a cost of US$19.5 M. The drill targets will be defined as ongoing results become available, but expansion and infill at Kakula West remains a priority, as well as additional exploration drilling planned to test targets located elsewhere within the Project.

1.16.3 Processing Plant

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

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

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

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

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

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

2 INTRODUCTION

2.1 Ivanhoe Mines Ltd

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

Ivanhoe currently has three key assets: (i) the Kamoa-Kakula Project (the Project); (ii) the Platreef Project, and (iii) the Kipushi Project. In addition, Ivanhoe holds interests in prospective mineral properties in the DRC, South Africa, Gabon, and Australia, including a land package of ~9,000 km^2 in the Central African Copperbelt with drill-ready grass-roots prospects.

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

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

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

2.2 Terms of Reference

The Kamoa-Kakula 2018 Resource Update includes restatement of the Kamoa-Kakula 2017 Development Plan, which includes the Mineral Reserve on Kansoko from the Kamoa 2017 PFS and the results of the preliminary economic assessment (PEA) from the Kakula 2017 PEA. The Mineral Reserve in the Kamoa 2017 PFS remains valid. Further study work is currently incomplete and has not determined any results that require material changes to the Kamoa 2017 PFS.

The Kamoa-Kakula 2018 Resource Update is an independent NI 43-101 Technical Report (the Report) prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) for Ivanhoe for the Project located in the DRC.

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

The following companies have undertaken work in preparation of the Report:

- OreWin: Overall Report preparation, Kakula 2017 PEA analyses, Kakula and Kamoa North underground mining, Kamoa-Kakula combined production schedules, and financial models.
- Amec Foster Wheeler: Geology, drillhole data validation, and Mineral Resource estimation for Kamoa and Kakula.
- SRK Consulting Inc.: Kansoko PFS Mine geotechnical recommendations.
- Stantec Consulting and KGHM Cuprum: Underground mine planning and design. MDM/Amec Foster Wheeler: Process and infrastructure.
- Golder Associates: paste backfill, hydrology, hydrogeology, and geochemistry.
- Epoch Resources (Pty) Ltd: Tailings Storage Facility (TSF).
- Kamoa Copper SA property description and location, ownership, mineral tenure, environmental studies, permitting and social and community and marketing.

This Report uses metric measurements. The currency used is U.S. dollars (US$).

2.3 Qualified Persons

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

- Bernard Peters, B. Eng. (Mining), FAusIMM (201743), employed by OreWin as Technical Director - Mining was responsible for: Sections 1.1, 1.3, 1.4, 1.12, 1.13, 1.13.5, 1.14.1 to 1.14.3, 1.14.5, 1.15.1, 1.15.3, 1.16.1; Section 2; Section 3; Section 4, Section 5; Section 10.8, Section 16.3; Section 19; Section 20; Sections 21.1, 21.6 to 21.10; Section 22; Section 23; Sections 24.1 to 24.5, 24.7.8, 24.8; Sections 25.1, 25.3, 25.5; Section 26.1; Section 27.
- Dr. Harry Parker, SME Registered Member (2460450), Technical Director, Amec Foster Wheeler a division of Wood plc was responsible for: Sections 1.2, 1.5 to 1.9, 1.11, 1.15.2, 1.16.2; Section 2; Section 3; Section 6; Section 7; Section 8; Section 9; Sections 10.1 to 10.6, 10.10 to 10.12; Sections 11.1 to 11.3, 11.5 to 11.12; Section 12, Section 14; Section 25.2; Section 26.2; Section 27.
- Gordon Seibel, SME Registered Member (2894840), Principal Geologist, Amec Foster Wheeler a division of Wood plc was responsible for: Sections 1.2, 1.5 to 1.9, 1.11, 1.15.2, 1.16.2; Section 2; Section 3; Section 6; Section 7; Section 8; Section 9; Sections 10.1 to 10.6, 10.10 to 10.12; Sections 11.1 to 11.3, 11.5 to 11.12; Section 12, Section 14; Section 25.2; Section 26.2; Section 27.
- William Joughin, FSAIMM (55634), employed by SRK Consulting (South Africa) (Pty) Ltd as Principal Consultant, was responsible for: Section 2; Section 10.7: Section 16.1.
- Jon Treen P. Eng. (Mining), PEO (90402637), employed by Stantec Consulting International LLC as Mining Business Line Leader, was responsible for: Sections 1.13.1, 1.13.4, 1.15.4; Section 2; Section 3; Section 15; Section 16.2; Section 21.2; Section 25.4; Section 26.3; and Section 27.
- Dean David, FAusIMM(CP) (102351), Technical Director – Process, Amec Foster Wheeler, Mining and Metals, Australia West, was responsible for: Sections 1.10, 1.13.2, 1.13.3, 1.14.4, 1.16.3; Section 2.3 and 2.4; Section 10.9; Section 11.4; Section 13; Section 17; Section 18; Sections 21.3, 21.4, 21.5; Sections 24.6, 24.7; Section 26.4; Section 27.

2.4 Site Visits and Scope of Personal Inspection

Site visits were performed as follows:

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

Dr. Harry Parker visited the Kamoa-Kakula Project from 1 to 3 May 2009, from 27 to 30 April 2010, from 12–14 November 2012, and again from 17–19 January 2017. The site visits included presentations by Ivanhoe and African Mining Consultants' staff, inspection of core and surface outcrops, viewing drill platforms and sample cutting and logging facilities, and discussions of geology and mineralisation interpretations with Ivanhoe's staff. On his January 2017 visit, Dr. Parker checked drillhole locations, inspected drill core, and collected witness samples from the Kakula deposit.

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

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

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

Mr. Dean David visited the Kamoa project site from 27 to 30 April 2010, and again from 13 to 15 April 2011, where he conducted an inspection of core, sample cutting and logging areas, discussed geology and mineralisation interpretations with Ivanplats' staff, presented metallurgical test results and participated in selection of samples for upcoming metallurgical testwork programs.

Table 2.1 SRK Site Visits

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

2.5 Effective Date

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

- Effective date of the Report: 23 March 2018.
- Date of the database closure Kamoa Mineral Resource estimate: 23 November 2015.
- Date of the database closure Kakula Mineral Resource estimate: 26 January 2018.
- Date of drill information from the ongoing drill program at Kakula: 21 February 2018.
- Date of drill information from the 2016–2017 drill program at Kamoa and date of updated copy of the database: 27 November 2017. Information available from this data supply was used to validate the Kamoa geological model.
- Date of Mineral Resource estimate for Kamoa: 27 November 2017.
- Date of Mineral Resource estimate for Kakula: 23 February 2018.
- Date of the Mineral Reserve estimate for Kamoa; 28 November 2017.
- Date of the supply of legal information supporting mineral tenure: 23 March 2018.

2.6 Information Sources and References

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

3 RELIANCE ON OTHER EXPERTS

The QPs, as authors of Kamoa-Kakula 2018 Resource Update, have relied on, and believe there is a reasonable basis for this reliance, upon the following Other Expert reports as noted below. Individual QP responsibilities for the sections are listed on the Title Page.

3.1 Mineral Tenure

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

- Kamoa Copper SA: report on the Kamoa-Kakula Project Property Description and Location, January 2018.
- Emery Mukendi Wafwana & Associates, SCP., 2016: Validity of (i) The Exploration Permits relating to The Mining Project of Kamoa (ii) The Kamoa Exploitation Permits, (iii) The transfer of 45 of rest of The Kamoa Exploration Permits of Kamoa Copper SA to Ivanhoe Mines Exploration DRC SARL, addressed to Ivanhoe Mines Ltd.
- Andre–Dumont, H., 2013: Democratic Republic of the Congo: report prepared by McGuireWoods LLP in Bourassa M., and Turner, J., 2013 (eds): Mining in 31 jurisdictions worldwide 2013, Mining 2013, Getting the Deal Through, posted to http://www.mcguirewoods.com/news-resources/publications/international/miningdrcongo.pdf.
- Ivanhoe Mines DRC SARL, 2017, DRC Mining Code Review and Ministerial Decrees: unpublished internal email prepared by Corporate Affairs Ivanhoe Mines DRC SARL, 28 June 2017.

This information was used in Section 4.3 of the Report and Section 14.12 for assessment of reasonable prospects of eventual economic extraction.

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

- Kamoa Copper SA: report on the Kamoa-Kakula Project Property Description and Location, March 2018.

This information was used in Section 4 of the Report, and Section 14.14 for assessment of reasonable prospects of eventual economic extraction.

3.2 Surface Rights

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

- Kamoa Copper SA: report on the Kamoa-Kakula Project Property Description and Location, March 2018..

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

3.3 Environmental and Work Program Permitting

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

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

This information was used in Section 20 of the Report and Section 14.12 for assessment of reasonable prospects of eventual economic extraction.

3.4 Taxation and Royalties

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

- KPMG Services (Pty) Limited, 2016: Letter from M Saloojee, Z Ravat, and L Kiyombo to M Cloete and M Bos regarding Updated commentary on specific tax consequences applicable to an operating mine in the Democratic Republic of the Congo, dated 01 March 2016.
- Kamoa Copper SA, 2017: Kamoa Copper Project: unpublished letter prepared by Kamoa Copper SA for OreWin, 26 June 2017.
- Kamoa Copper SA: report on the Kamoa-Kakula Project Property Description and Location, March 2018.

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

4 PROPERTY DESCRIPTION AND LOCATION

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

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

Figure 4.1 Project Location Map

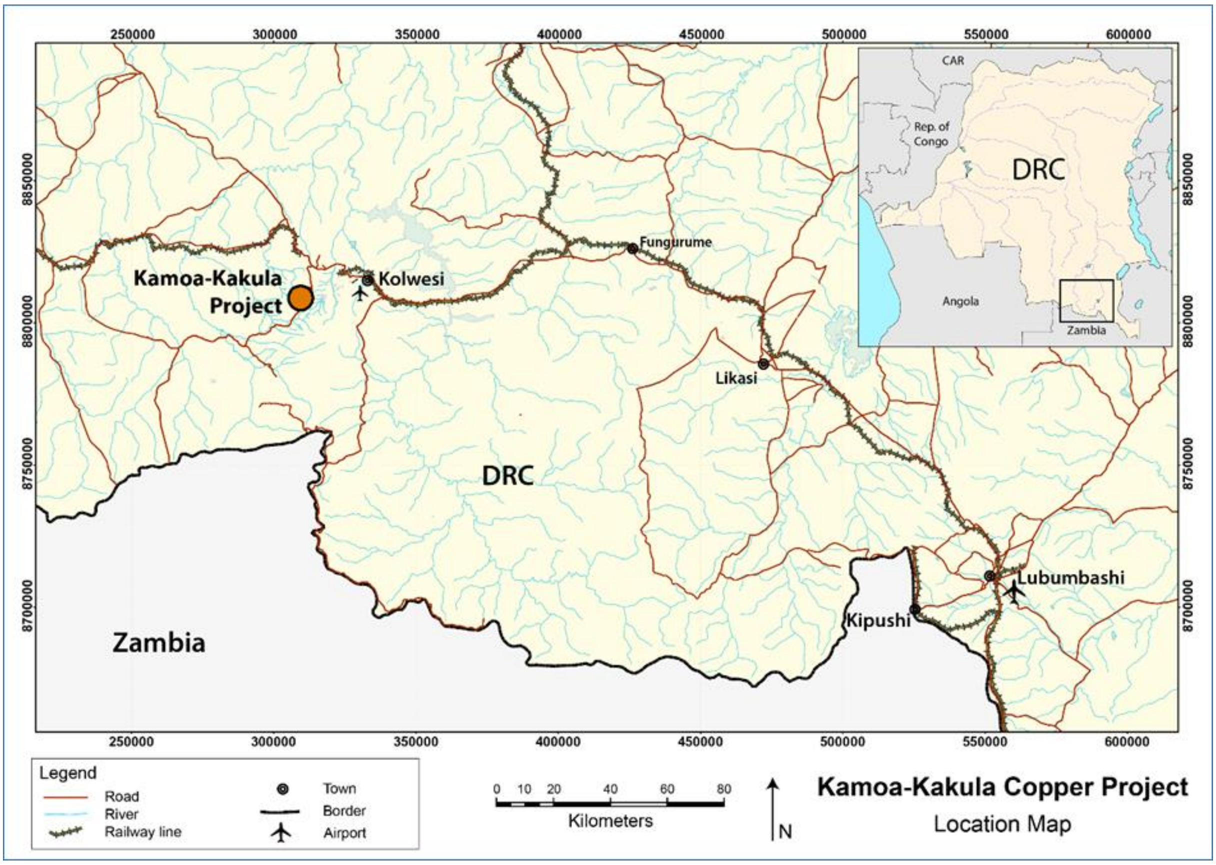


Figure by Ivanhoe, 2016.

4.1 Project Ownership

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

On 11 November 2016, Kamoa Holding Limited 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 Kamoa-Kakula 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
Kamoa-Kakula Project while Crystal River Global Limited holds an indirect 0.8% interest and the DRC holds a direct 20% interest in the Kamoa-Kakula Project.

The share transfer agreement provides, without limitation, that:

- Kamoa Holding Limited (Kamoa Holding) will transfer 300 Class A shares in the capital of Kamoa Copper SA – representing 15% of Kamoa Copper SA's share capital – to the DRC, in consideration for a nominal cash payment and other guarantees from the DRC summarised below. In addition, the DRC owns 100 non-dilutable Class B shares, representing 5% of Kamoa Copper SA's share capital.
- The parties agreed that the 300 Class A shares shall be non-dilutable until the earlier of (i) five years after the date of the first commercial production and (ii) the date on which the DRC ceases to hold all of its 300 Class A shares.
- Kamoa Holding undertakes to provide all shareholder loans to Kamoa Copper SA and/or procure the project financing from third parties for the development of the Kamoa-Kakula Project.
- Kamoa Holding Limited 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 US$659 million on 28 February 2018), and 100% of any financing of the project by third parties.
- The DRC confirmed that the Kamoa-Kakula Project will be developed with the support of the government of DRC and of its Ministry of Mines by Kamoa Copper 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 Kamoa-Kakula Project are at the date of the signature of the share transfer agreement valid and in good standing, without any defect and that Kamoa Copper SA's mining rights are not subject to any cancellation or to any litigation or dispute, whatsoever and recognised and guaranteed the peaceful enjoyment of its mining rights by Kamoa Copper SA.
- The DRC has also confirmed and guaranteed that the Kamoa-Kakula Project will not be subject to any taxes or duties other than those legally required by the applicable statutory and regulatory provisions.
- At Kamoa Copper SA's request and subject to the satisfaction of the applicable conditions, the DRC State shall provide its assistance to Kamoa Copper SA, its affiliates and subcontractors for the purpose of obtaining the advantages contemplated by the DRC's special law – No. 14/005 dated 11 February 2014, determining the tax, customs, parafiscal tax, non-tax revenues and currency exchange regime applicable to collaboration agreements and cooperation projects.

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

4.2 Property and Title in the Democratic Republic of the Congo

4.2.1 Introduction

A summary of the mining history of the Katangan region is presented below, and is adapted from André-Dumont (2013), and the 2002 DRC 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 Miniere du Haut Katanga in 1906 and first industrial production of copper in 1911, from l'Etoile (Ruashi), a very rich copper oxide deposit located a few kilometres from Lubumbashi. Just prior to 1960, the DRC was the world's fourth-largest producer of copper and supplied 55% of the world's cobalt from deposits in Katanga. Following independence from Belgium in 1960, production gradually decreased due to a combination of factors that included political unrest, political and social environments within the country, declining investment in infrastructure, and lack of capital (Goossens, 2009).

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

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

4.2.2 Mineral Property Title

The following summary on mineral title is adapted from André-Dumont (2013), the 2002 DRC Mining Code and available information on Law No. 18/001 (the draft law amending the 2002 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 (Law No. 007/2002) dated 11 July 2002 (the 2002 Mining Code), as amended by Law No. 18/001 dated 09 March 2018 (the 2002 Mining Code, as amended by Law No. 18/001, is hereafter referred to as the 2018 Mining Code), which is clarified by the Mining Regulations enacted by Decree No. 038/2003 of 26 March 2003 (the 2003 Mining Regulations). The 2003 Mining Regulations should be amended before 09 June 2018. A review process in this respect, in consultation with the main mining operators in DRC, is ongoing. The legislation incorporates environmental requirements.

Insofar as on the date of this report, the above mentioned Law No. 18/001 has not been published in the DRC official gazette and as no official version of this Law was published by any DRC authority so far, this report, including economic analysis, is based upon the 2002 Mining Code. On the basis of the unofficial draft version that have circulated, we nevertheless understand that Law No. 18/001 includes significant changes to the investment framework for mining operators in the DRC, such as royalties, taxation, stability, local content requirements and other technical matters, including concerning the regime applicable to mining rights.

The drafting of mining regulations for the implementation of this new law has not yet been finalised and a consultation process is ongoing with mining operators in the DRC in this respect.

Following a meeting between President Joseph Kabila Kabange held on 07 March 2018, senior members of the government and senior representatives of international mining companies that have operations in DRC, detailed discussions are ongoing with the aim of resolving, in a fair and equitable manner, the mining industry’s concerns with the 2018 Mining Code.

The Ministry of Mines supervises the Cadastre Minier (DRC mining registry), the Directorates of Mines and Geology and the department in charge of the protection of the mining environment in the DRC under the 2002 Mining Code. The main administrative entities in charge of regulating mining activities in the DRC as provided by the 2002 Mining Code are, without limitation, the following:

- The President of the DRC, who is notably responsible for enacting the Mining Regulations for the implementation of the 2002 Mining Code. The President of the DRC exercises his rights by decrees to be published in the DRC Official Gazette.
- The Minister of Mines, who has notably jurisdiction over the granting, refusal and withdrawal of mining rights. He exercises its powers by ministerial orders to be published in the DRC Official Gazette.
- The Cadastre Minier, it is a public entity supervised by the Minister of Mines and the Minister of Finance that conducts in particular administrative proceedings concerning the application for, and registration of, mining rights, as well as the withdrawal, cancellation and expiry of those rights.
- The Directorate of Mines, which is notably responsible for controlling and inspecting mining activities with regard to safety, health, work conduct, production, transport, marketing and social matters in accordance with the 2002 Mining Code and the 2003 Mining Regulations.
- The Department in charge of the protection of the mining environment, which is notably responsible for definition and implementation of the mining regulation concerning environmental protection and the technical examination of the environmental impact studies, and environmental management plans. It is also notably responsible for controlling the implementation of the environmental mitigation and rehabilitation measures by the holder of mining rights and verify their efficiency.

Under the 2002 Mining Code, the mining rights are Exploration Permits, Exploitation Permits, Small Scale Exploitation Permits, and Tailings Exploitation Permits.

There are no distinctions between mining rights that may be acquired by DRC domestic parties and those that may be acquired by a foreign company.

Foreign companies must elect domicile with an authorised DRC domestic mining and quarry agent (mandataire en mines et carrières) and act through this intermediary. The mining or quarry agent acts on behalf of, and in the name of, the foreign national or foreign legal entity with the mining authorities, mostly for the purposes of communication.

Foreign companies need not have a domestic partner, but a company that wishes to obtain an Exploitation Permit must transfer 5% (non-dilutable and free of any charge) of the shares in the share capital of the applicant company to the DRC State.

The 2002 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 2003 Mining Regulations. This grid defines uniform quadrangles, or cadastral squares, typically 84.955 ha in area, which can be selected as a "Perimeter" to a mining right. A perimeter under the 2002 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 2003 Mining Regulations.

The geographical location of the Perimeter is identified by the coordinates at the centre of each quadrangle which make up the Perimeter.

Perimeters are exclusive, and may not overlap subject to specific exceptions listed in the 2002 Mining Code and 2003 Mining Regulations. Perimeters are indicated on 1:200,000 scale maps that are maintained by the Cadastre Minier.

Within two months of issuance of a mining or quarry 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, and placing a permanent post indicating the name of the holder, the number of the title and that of the identification of the survey marker.

4.2.3 DRC Mining Code Review and Local Content Requirements

When the 2002 Mining Code was introduced, the DRC Government indicated that after a 10-year period, a review would be undertaken. In February 2013, a draft law on the revision of the 2002 Mining Code was circulated by the DRC Minister of Mines. The proposed amendments to the 2002 Mining Code contained in the draft law included more onerous formalities and conditions for obtaining mining rights. However, in February 2016, the DRC Minister of Mines announced that the current 2002 Mining Code would be retained.

On 21 March 2017, the Interinstitutional meeting nevertheless recommended the amendment of the 2002 Mining Code. On 6 June 2017, the Minister of Mines presented to the Parliament a draft bill on the proposed amendment of the Mining Code. It was declared receivable and was examined by three commissions of the Congolese Parliament in order to prepare the debates. The Chamber of Mines expressed some concerns from mining companies and investors, including, without limitation, concerning the increased tax burden for mining operators and the proposed change to the stability clause of the 2002 Mining Code.

In spite of those concerns, the DRC National Assembly adopted the amendment of the 2002 Mining Code on 8 December 2017. It was then sent to the DRC Senate for its adoption before its promulgation by the President of DRC.

On 24 January 2018, the amendment of the 2002 Mining Code was passed by the DRC Senate. It was then sent to committee to be harmonized before the final version is sent to the President of DRC for promulgation. In spite of allegations in this respect, no evidence was found concerning the respect of the requirement set out by Article 35 of DRC Constitution concerning a vote of the final version of the draft law by the two chambers of Parliament before the communication of the draft version to the President of DRC.

On 7 March 2018, a meeting took place between President Joseph Kabila Kabange, senior members of the government and senior representatives of international mining companies that have operations in the DRC to discuss the draft law amending the 2002 Mining Code. In this meeting, the President gave an assurance that once the new law had been promulgated, the questions raised by the mining industry would be resolved through transitional arrangements, mining regulations and respect of agreements and guarantees.

On 9 March 2018, Law No. 18/001 amending the 2002 Mining Code was promulgated. As of the date of this report, Law No. 18/001 has not been published in the DRC official gazette and the drafting of mining regulations for the implementation of this new law has not been finalised. Detailed discussions are ongoing with the aim of resolving, in a fair and equitable manner, the mining industry's concerns with the 2018 Mining Code.

As soon as Law No. 18/001 will be published and as there is more clarity on the mining regulations governing the implementation of the 2018 Mining Code, as well as potential adaptations to the 2018 Mining Code, if any, a thorough review will be performed to clarify its implications for the Kamoa-Kakula Project with regard to the commitment made in the share transfer agreement dated 11 November 2016. Information in this report, including economic analysis, is therefore based upon the 2002 Mining Code until such clarity is provided.

During 2013, the DRC Minister of Mines passed two ministerial orders.

The first, dated 5 April 2013 and adopted together with the Minister of Finance, bans the export of copper and cobalt concentrates, and includes a reduced moisture content requirement for concentrates intended for export. It also limits the costs that are deductible for the determination of the mining royalty.

However, annual moratoriums were adopted concerning the ban of export of copper and cobalt concentrates. On 30 December 2016, an interministerial order was adopted by the Minister of Mines and Finance that clarified that a moratorium extended until the definitive resolution of the energy deficit is granted to all mining operators that produce copper and cobalt concentrates.

The second order, dated 17 April 2013, requires mining operators to only use Congolese businesses for subcontracting their direct mining activities (including development and construction works) as well as for connected and ancillary activities. The Congolese business that is the beneficiary of a subcontracting agreement may nevertheless use, where required, for the performance of the subcontracted activities, exterior expertise or a qualified foreign company.

However, there has been objection to this order that argues it is contrary to the 2002 Mining Code and in particular to its article 273 of which provides that mining companies holding mining rights are free to import goods, services as well as funds necessary to their activities subject to giving priority to Congolese businesses for all contracts in relation to the mining project, at equivalent conditions in terms of quantity, quality, price, delivery deadlines and payment.

In 2014, a new order was also adopted on the basis of the necessity to grant priority to Congolese industries, small and medium businesses for the supply of services, procurement in goods and other inputs of local production for the needs of mining companies performing their activities in DRC. Pursuant to this order, mining companies must use Congolese industries, small and medium businesses for services, the supply of goods and the procurement of inputs and other consumables, including lime (chaux) and its derivatives (dérivés) and cement. However, if the needs expressed by the mining companies exceed the capacity of the Congolese industries, small and medium businesses, mining companies are authorised to import the goods, inputs and other consumables in order to fill the insufficiency of their production. There was also some objection that argues that the validity of this order could potentially be challenged.

Finally, in 2017, Law No. 17/001 dated 8 February 2017 setting out the rules applicable to subcontracting in the private sector (the Subcontracting Law) was adopted and determines the rules applicable to subcontracting between private law individual or legal entities. The Subcontracting Law does not explicitly exclude the mining sector from its scope. However, in an analysis of the Subcontracting Law, Hubert André-Dumont, a respected legal author, familiar with DRC law, consider that on the basis of the exception set out by Article 2 of the Subcontracting Law, the mines sector is excluded from the scope of the Subcontracting Law insofar as pursuant to this provision, subcontracting concerns all sectors of activities, subject to legal provisions governing specific sectors of activities, the mining sector being governed by specific legislations, including provisions governing subcontracting. There are nevertheless consistent expectations from civil society and local authorities to see the Subcontracting Law implemented to the mining sector in spite of the exception set out by the above-mentioned provision.

In addition, there are in the Subcontracting Law some new requirements applicable to all companies such as, for instance, an obligation to publish each year the turnover realised with subcontractors as well as the list of these subcontractors and to implement, within the Companies, a training policy enabling Congolese to acquire the technicity and qualification required for the performance of some activities. Otherwise, the Subcontracting Law appears to mainly govern the relationships between the main contractor (entreprise principale) and the subcontractors as defined by the Subcontracting Law (different from the definition set out by the 2002 Mining Code).

Pursuant to the Subcontracting law, subcontracting, as defined by this law, is now an activity reserved to businesses with Congolese capital (capitaux congolais), promoted by Congolese and having their head office in DRC. However, when there is non-availability or non-accessibility of the above expertise and subject to providing evidence to the relevant authority, the main contractor is authorised to enter into an agreement with any other Congolese or foreign business for a maximum duration of six months or to create a Congolese company. The sectorial Minister or local authority must be informed previously. Subcontracting is limited to a maximum of 40% of the global value of a contract. In addition, the main contractor is not authorized to oblige the subcontractor to totally prefinance the cost of the subcontracted operation or activity and must pay, before the beginning of the works, an advance payment covering at least 30% of the subcontracting contract. Any subcontracting above approximately USD 62,000 require a tendering process (appel d'offres). As from 15 March 2018, foreign businesses having subcontracting contracts must create Congolese companies to perform those contracts and Congolese businesses having valid subcontracting contracts must comply with the provisions of the Subcontracting Law. Fines for non-compliance with the Subcontracting Law are significant. However, as criminal law must be interpreted restrictively, the rules applicable to the Kamoa-Kakula Project resulting from the Subcontracting Law should, from a legal perspective, be very limited.

Subject to its publication and to further analysis, we understand that the 2018 Mining Code increases the requirements applicable for the selection by the holder of mining rights of its subcontractors, as defined under the 2018 Mining Code and could require the implementation of the Subcontracting Law to the mining sector.

Further analysis will be performed once the outcome of the ongoing dialogue between mining operators in DRC and the Government will be known to ensure that the Kamoa-Kakula Project and its contractors and subcontractors comply with applicable legal requirements.

4.2.4 Exploration Permits

An Exploration Permit, as defined in the 2002 Mining Code, grants to its holder the exclusive right to perform, within the perimeter over which it is established and during its validity period, exploration works of mineral substances classified in mines for which the Exploration Permit was granted and associated substances if the holder applies for the extension of the Exploration Permit to these substances.

At the time of Exploration Permit application, a holder specifies which minerals, and/or precious stones, to which the permit will apply. The 2002 Mining Code makes provision for this list to be formally extended to include additional commodities where research results justify such extensions.

Under the 2002 Mining Code, permits are granted for all minerals other than precious stones for a term of five years, and are renewable twice for a period of five years each renewal.

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

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

Renewal applications automatically require a 50% ground relinquishment with each application. If an entire Exploration Permit, or part of an Exploration Permit, is converted to an Exploitation Permit, the portion that has been converted is no longer subject to ground relinquishment requirements.

In other respects, under the 2002 Mining Code the holder of an Exploration Permit is authorised to take samples of the mineral substances within the Perimeter indicated on the Exploration Permit for analysis or industrial assays in the laboratory or plant of holder's choice.

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

An Exploration Permit holder can convert part of the permit into an Exploitation Permit, or a small-scale Exploitation Permit, and still maintain the rights to explore on the remainder of the Exploration Permit, subject to conditions laid out in the 2002 Mining Code.

4.2.5 Exploitation Permits

Exploitation permits are valid for 30 years, renewable for 15-year periods until the end of the mine's life, if conditions laid out in the 2002 Mining Code are met.

Granting of an Exploitation Permit is dependent on a number of conditions that are defined in the 2002 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 planned financing and justification of their probable availability.
3. Obtain in advance the approval of the project's Environmental Impact Studies (EIS) and the Project Environmental Management Plan (PEMP).
4. Transfer to the DRC State 5% of the shares in the share capital of the company applying for the Exploitation Permits. These shares are free of all charges and cannot be diluted.

The Exploitation Permit, as defined in the 2002 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 EIS and the PEMP.
4. Dispose (disposer), transport and freely market his 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 2002 Mining Code, or when the deposit that is being mined is exhausted.

For renewal purposes under the 2002 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 2002 Mining Code.
- Demonstrated that the deposit is not exhausted by updating the feasibility study in accordance with the laws of the DRC.
- Demonstrated the existence of the financial resources required to continue to carry out his 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 planned and the justification of its probable availability.
- Obtained the approval of the update of the EIS and PEMP.
- Undertaken in good faith to actively carry on with his exploitation.

Under the 2002 Mining Code, the sale of mining products which originate from the Exploitation Permit is "free", meaning that the holder of an Exploitation Permit may sell any licensed products to a customer of choice, at "prices freely negotiated".

Under the 2002 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 2002 Mining Code). Levies are defined on a per hectare basis, and increase on a sliding scale for each year that the mining title is held, until the third year, after which the rate remains constant. In this Report, this levy is referred to as a "surface rights fee".

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

4.2.6 Surface Rights Title

The following summary on surface rights title is adapted from André-Dumont (2008, 2011), and the 2002 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 2002 Mining Code provides that subject to the potential rights of third parties over the relevant soil, the holder of an exploitation mining right has, with the authorisation of the Governor of the relevant Province, after opinion from the relevant department of the Administration of Mines notably within the perimeter of the mining right, the right to occupy the parcels of land required for its activities and the associated industries, including the construction of industrial facilities, dwellings and facilities with a social purpose, to use underground water, the water from non-navigable, non-floatable watercourses, notably to establish, in the context of the concession of a waterfall, an hydroelectric power plant aimed at satisfying the energy needs of the mine, to dig canals and channels, and establish means of communication and transport of any type. Kamoa Copper SA was granted with such an authorisation from the Governor of the Province on 23 July 2014.

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

4.2.7 Environmental Regulations

The following summary on environmental regulations is adapted from André-Dumont (2008, 2011), and the 2002 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 environmental impact study (EIS) to be filed, together with the project environmental management plan (PEMP) to be approved by the relevant authorities.

On approval, the applicant must provide security for rehabilitation. The security can be provided by means of a bank guarantee. Funds posted as security are not at the disposal of the Department in charge of the protection of the mining environment of the Ministry of Mines and are to be used for the rehabilitation of a mining site. Kamoa Copper SA complied with its obligation in this respect in accordance with the instalments set out in the approved updated EIS and is in the process of securing and filing the additional financial guarantee required for 2018 .

Exploration Permit

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

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

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

Exploitation Permit

Environmental obligations for conversion of an Exploration Permit to an Exploitation Permit under the Mining Code require the preparation of an EIS and a PEMP.

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

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

The Mining Code requires an environmental audit every two-year period as from the date of approval of the initial project EIS. Such an environmental audit will start very shortly in accordance with applicable law.

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

Upon mine closure, shafts must be filled, covered or enclosed, and a certificate obtained confirming compliance with environmental obligations under the terms of the approved EIS and PEMP.

4.2.8 Royalties

A company holding an exploitation permit is subject to mining royalties. The mining royalty is due upon the sale of the product and is calculated at 2% of the price received of non-ferrous metals sold less the costs of transport, analysis concerning quality control of the commercial product for sale, insurance and marketing costs relating to the sale transaction.

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

Subject to verification of the published version of the law promulgated on 09 March 2018 amending the 2002 Mining Code, we understand that the 2018 Mining Code includes a proposed increase of the mining royalty for copper from 2% to 3.5%. The basis of this royalty should also change to become the gross commercial value at the time of exit of the marketable product from the site. As of the date of this report, the 2018 Mining Code has not been published in the DRC official gazette and the drafting of mining regulations for the implementation of this new law has not been finalised. Detailed discussions are ongoing with the aim of resolving, in a fair and equitable manner, the mining industry's concerns with the 2018 Mining Code.

A thorough analysis on the implications of the 2018 Mining Code will be performed as soon as there is more clarity on 2018 Mining Code and on the mining regulations governing the implementation of the 2018 Mining Code.

4.3 Mineral Tenure

The Kamoa-Kakula Project consists of the Kamoa Exploitation Licences (exploitation permits 12873, 13025, and 13026 which cover an area of 397.6 km^2) and one exploration licence (Exploration Permit 703 covers an area of 12.74 km^2). A mineral tenure summary table is provided in Table 4.1 and the mineral tenure locations are as indicated in Figure 4.2. The Exploitation Permits were surveyed and boundary marked together with the Cadastre Minier. Exploration Permits are delineated by latitude/longitude co-ordinates and do not require survey.

Table 4.1 Permit Summary Table

Exploitation Permit (PE) No.	Grant Date	Expiry Date	Mineral/Metal Rights Granted	Number Cadastral Squares (Quadrangles)	Area (km^2)
12873	20 Aug 2012	19 August 2042	Silver, Bismuth, Cadmium, Cobalt, Copper, Iron, Germanium, Nickel, Gold, Palladium, Platinum, Lead, Rhenium, Sulphur and Zinc.	62	52.7
13025	20 Aug 2012	19 August 2042	Silver, Bismuth, Cadmium, Cobalt, Copper, Iron, Germanium, Nickel, Gold, Palladium, Platinum, Lead, Rhenium, Sulphur and Zinc.	204	173.3
13026	20 Aug 2012	19 August 2042	Silver, Bismuth, Cadmium, Cobalt, Copper, Iron, Germanium, Nickel, Gold, Palladium, Platinum, Lead, Rhenium, Sulphur and Zinc.	202	171.6
Sub Total					**397.6**
703 (Exploration Permit)	11 Nov 2003	10 May 2020	Base, Precious, Platinum Group Metals, Pegmatite Minerals, Diamonds and Gemstones	15	12.7
Sub Total					**12.7**
Total					**410.3**

Figure 4.2 Project Tenure Plan

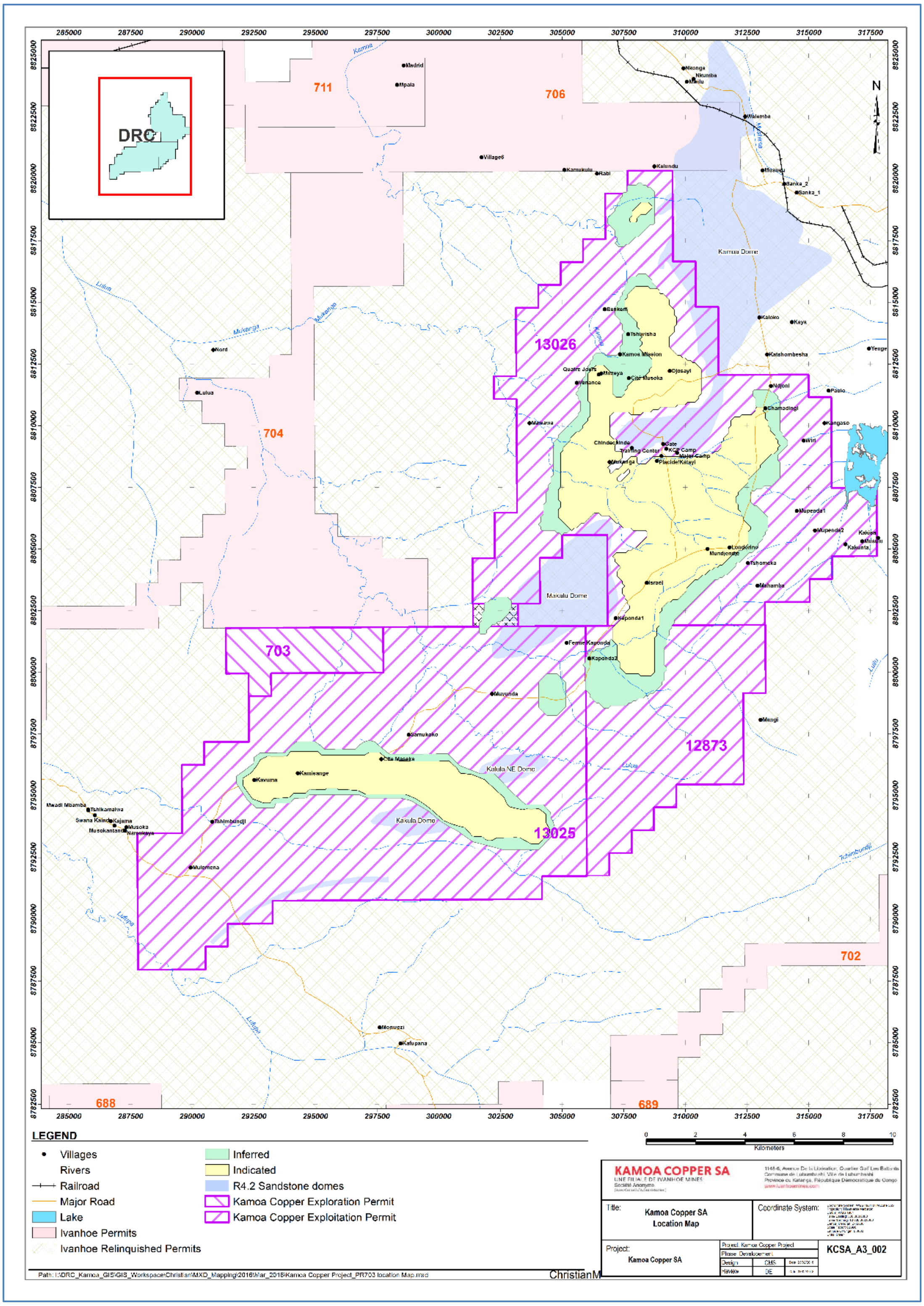


Figure by Kamoa Copper SA, 2018.

Ivanhoe advised the QPs that Ivanhoe had pro-rata paid the required surface fees for the Exploitation Permits to the DRC government, as this pre-payment was a pre-condition of grant of the permits. The surface rights fee is due by 31 March of each year; land taxes are due by 1 February of each year. Ivanhoe advised the QPs that the required land tax payments for 2017 were made for the three Exploitation Permits and Exploration Permit 703, and that Ivanhoe has paid the required fees for 2018.

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

4.4 Surface Rights

At the effective date of this Report, Kamoa Copper SA holds no surface rights in the Project area but is authorised to occupy the parcels of land required for its activities. Investigations with local administrations should be performed to clarify whether or not there are any holder of surface rights enforceable against third parties within the area of planned infrastructure. Land access for the exploration programmes completed to date has typically been negotiated without problems. Where compensation has been required for exploration activities, compensation has followed DRC 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 km2 required for the Kansoko mine as required by the DRC law to enable the company to occupy this land. A similar process is in progress for Kakula, including the planned resettlement of 45 households surveyed in the Kakula footprint. Once the compensation and resettlement is complete, Kamoa Copper SA will apply for prohibition areas (zones d'interdiction) where the activities and/or circulation of third parties will be prohibited for the areas required for the Kansoko and Kakula surface infrastructure that give the company the full legal right to occupy the relevant area and prevent any other parties occupying or entering the area.

4.5 VAT Exoneration

Holders of mining rights are normally entitled to exoneration for import duties and import VAT for all materials and equipment imported for construction of a mine and related infrastructure. Kamoa Copper SA has successfully received such exoneration in the past and expects to receive such exoneration for most imports for project construction.

4.6 Property Agreements

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

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

5.1.1 Air

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

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

5.1.2 Road

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

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

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

5.1.3 Rail

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

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

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

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

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

5.2 Climate

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

5.3 Local Resource and Infrastructure

The Project is currently isolated from public infrastructure. Infrastructure on-site is currently limited to support for the exploration activities.

Exploitation of the Kamoa and Kakula deposits will require building a greenfields project with attendant infrastructure.

Processing infrastructure exists in the Kolwezi mining district, but it is unknown whether this could be utilised by the Project.

5.4 Power

Power for the Kamoa-Kakula Project is planned to be sourced from three hydro power plants: the Koni, Mwadingusha and Nzilo 1 power stations. All three require refurbishing. The three plants combined could produce over 200 MW. Prior to completion of the refurbishments, development, and construction activities at the Kamoa deposit will be powered by electricity sourced from the grid and on-site diesel generators.

In June 2011, Ivanhoe signed a Memorandum of Understanding (2011 MOU) with the DRC's state-owned power company, SNEL. The 2011 MOU led to the signing of a pre-financing agreement with SNEL in June 2012 under which SNEL granted Ivanhoe an exclusive right to conduct full rehabilitation on the Mwadingusha and Koni plants following completion of a feasibility report. A study to rehabilitate the Mwadingusha and Koni power plants was carried out by Stucky Ltd in 2013 (Stucky Report). As well as the plant refurbishment, the alignment for the new high-voltage line to the Kamoa site is also required for power supply to the Project. This line is planned to be used at a reduced voltage during the construction phase and at the full rated voltage for production in 2018.

In 2013, Ivanhoe signed an additional Memorandum of Understanding (2013 MOU) with SNEL to upgrade a third hydroelectric power plant, Nzilo 1. Ivanhoe and SNEL plan to conduct a Feasibility Study to assess the scope of work and cost of restoration. It is proposed to upgrade the Nzilo 1 hydroelectric power plant to its design capacity of 111 MW.

5.5 Physiography

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

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

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

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

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

5.6 Comments on Section 5

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

6 HISTORY

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

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

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

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

The Kansoko Mine has a Mineral Reserve that was previously stated in the Kamoa 2016 Prefeasibility Study (Kamoa 2016 PFS). The base case described in the Kamoa 2016 PFS is the construction and operation of an underground mine, concentrator processing facilities, and associated infrastructure. The base case mining rate and concentrator feed capacity is 3 Mtpa. The production rate was increased to 6 Mtpa and mining methods changed for this Mineral Reserve update, Kamoa 2017 PFS.

The Kamoa 2016 Resource Technical Report was filed in November 2016 that included a first-time resource estimate for the Kakula deposit. In January 2017 the Kakula 2016 PEA was filed. The Kakula 2016 PEA included an analysis of the Kakula deposit as a standalone operation and a combined operation that is made up of the separate operations at the Kansoko Mine and the Kakula Mine at the Kakula deposit.

The Kakula 2017 Resource Update was released in a Technical Report in June 2017, this was followed by the Kamoa-Kakula 2017 Development Plan which was filed in January 2018. The Kamoa-Kakula 2017 Development Plan included an update of the Kamoa Mineral Reserve and updates of the PEA on the Kakula Mineral Resource. The production rate assumption at each deposit has increased from 4 Mtpa to 6 Mtpa, and the total combined production rate has increased from 8 Mtpa to 12 Mtpa.

The Mineral Reserves for the Kamoa 2017 PFS increased as a result of an increase in production rate through a change to the controlled convergence room-and-pillar mining method.

OreWin

7 GEOLOGICAL SETTING AND MINERALISATION

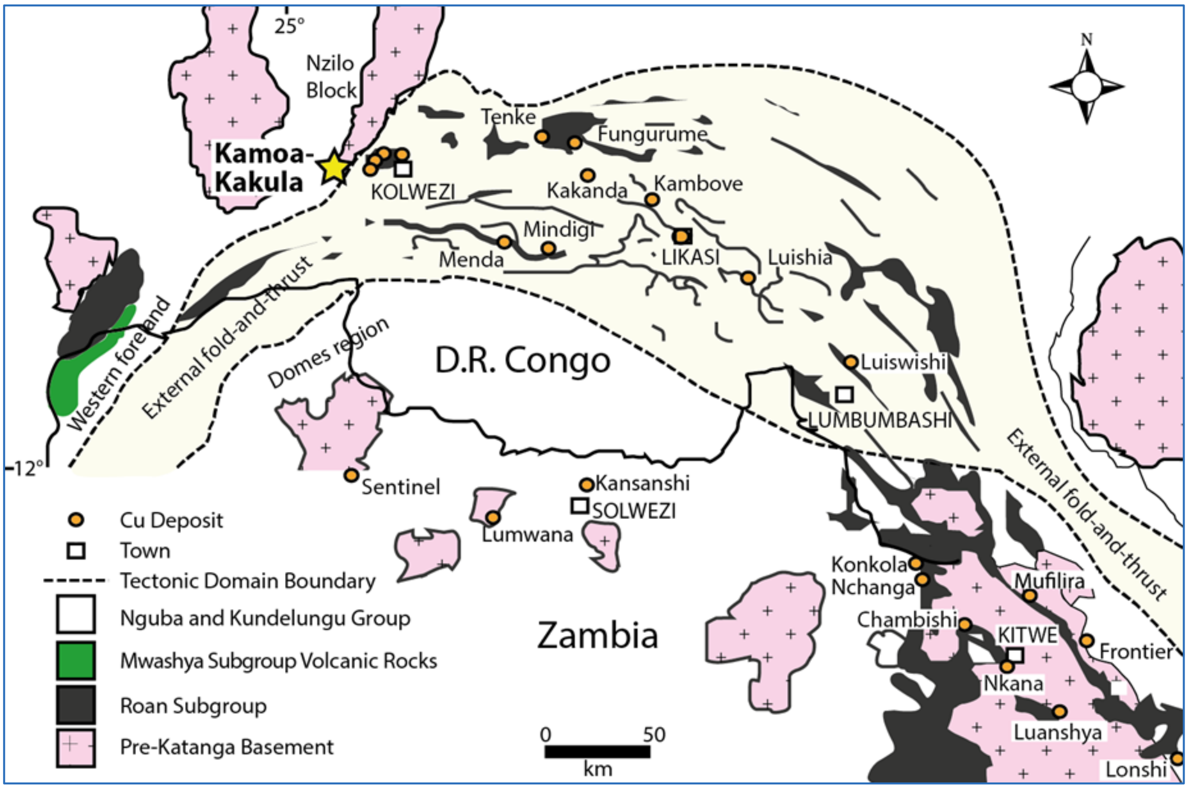
The discussion in this section has been prepared from published papers on regional geology as cited, particularly Schmandt et al (2013), and is also based on discussions with, and presentations made by, Ivanhoe personnel (David Broughton, David Edwards, and George Gilchrist) and African Mining Consultants (Thomas Rogers and Steffen Kalbskopf).

7.1 Regional Geology

The metallogenic province of the Central African Copperbelt is hosted in metasedimentary rocks of the Neoproterozoic Katanga Basin. The lowermost sequences were deposited in a series of restricted rift basins that were then overlain by laterally extensive, organic rich, marine siltstones and shales. These units ("Ore Shale") contain the bulk of the ore deposits within the Copperbelt (the Kamoa deposit is, however, an exception to this). This horizon is overlain by what became an extensive sequence of mixed carbonate and clastic rocks of the Upper Roan Group (Selley et al., 2005). The Roan Group now forms a northerly-directed, thin-skinned thrust-and-fold orogenic system, the Lufilian Arc, which resulted from the convergence of the Congo and Kalahari cratons (Figure 7.1). The metallogenic province is divided into two distinct districts, the Zambian and Congolese or Katangan Copperbelts.

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

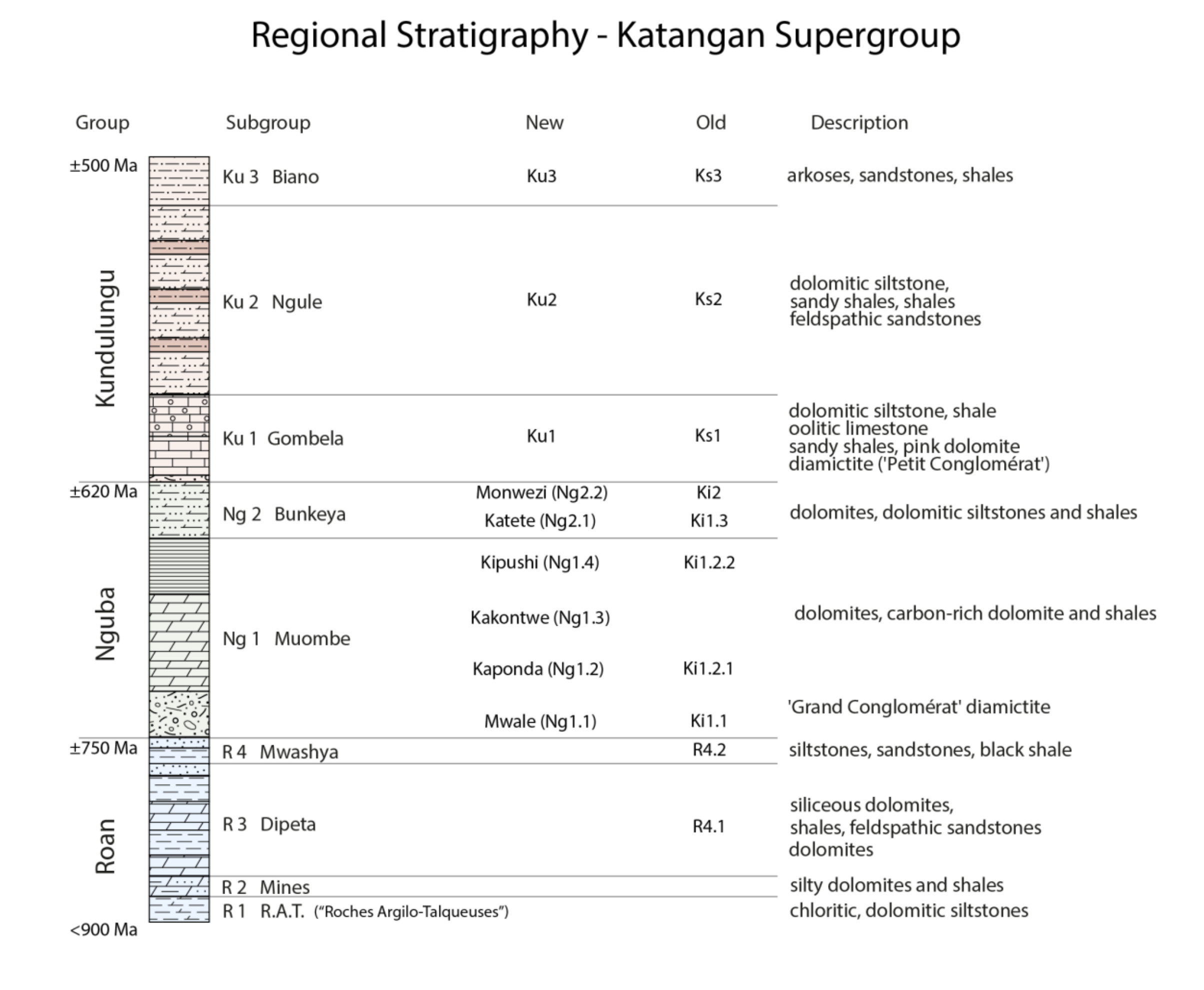
Figure 7.1 Geological Setting Central African Copperbelt



Source: Adapted from Schmandt et al (2013).

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

Figure 7.2 Stratigraphic Sequence, Katangan Copperbelt



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

The metasedimentary rocks of the Roan Group were deposited in an environment that was initially terrestrial in character, but evolved to a marine character during a regional transgression. In the basal Roan Group, temporarily anoxic conditions in a lagoonal to mudflat environment prevailed, giving rise to intercalations of evaporite-bearing rocks in the siliclastic–carbonate successions. Extensive evaporite deposits are interpreted to have formed during Roan time, but are no longer present, probably as a result of erosion/dissolution.

The tectonic setting of the Roan Group is somewhat uncertain due to orogenic overprinting. The most common interpretation is that Roan Group sediments accumulated in fault-bounded sub-basins (R1), which evolved from a continental rift basin filled by syn rift siliclastic rocks, to a laterally extensive carbonate platform (R2, R3).

Mineralisation in the majority of the Katangan Copperbelt orebodies such as at Kolwezi and Tenke–Fungurume (Figure 7.1) is hosted in the Mines supergroup (R2). The mineralisation at Kamoa differs from these deposits in that it is located in the Grand Conglomerat unit (Ki1.1) at the base of the lower Kundelungu Group.

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

7.1.1 Lufilian Orogeny

The Roan Group and succeeding Lower Kundulungu Group record two distinct rifting stages resulting from early Neoproterozoic extension. Mafic igneous rocks found within these have been interpreted as evidence for continued rifting throughout the deposition of the Roan and Lower Kundulungu sediments (Kampunzu et al., 2000). The Upper Kundulungu and younger formations were deposited in the succeeding foreland basins related to Pan African orogenesis.

The Katangan basin was inverted during the Pan-African Lufilian orogeny, from approximately 580 Ma to 500 Ma. The Lufilian Arc can be divided into subregions, of which the Katangan (Congo) Copperbelt in the DRC belongs to an outer terrane of the arc, the External Fold and Thrust Belt. This terrane is considered to be composed of a stack of thin-skinned and generally north-verging folds and thrust sheets. The thrust sheets occur together with megabreccias that may have a tectonic origin. Alternate explanations for the breccias include sedimentary sources, or salt tectonism.

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

West of the External Fold and Thrust Belt, in north-western Zambia, the Western Foreland (Foreland) comprises weakly deformed, autochthonous, siliciclastic and volcanic rocks of Roan age, and overlying Lower Kundelungu diamictite (Key et al, 2001). The Kamoa-Kakula Project area is considered by Ivanhoe to form part of the Foreland.

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

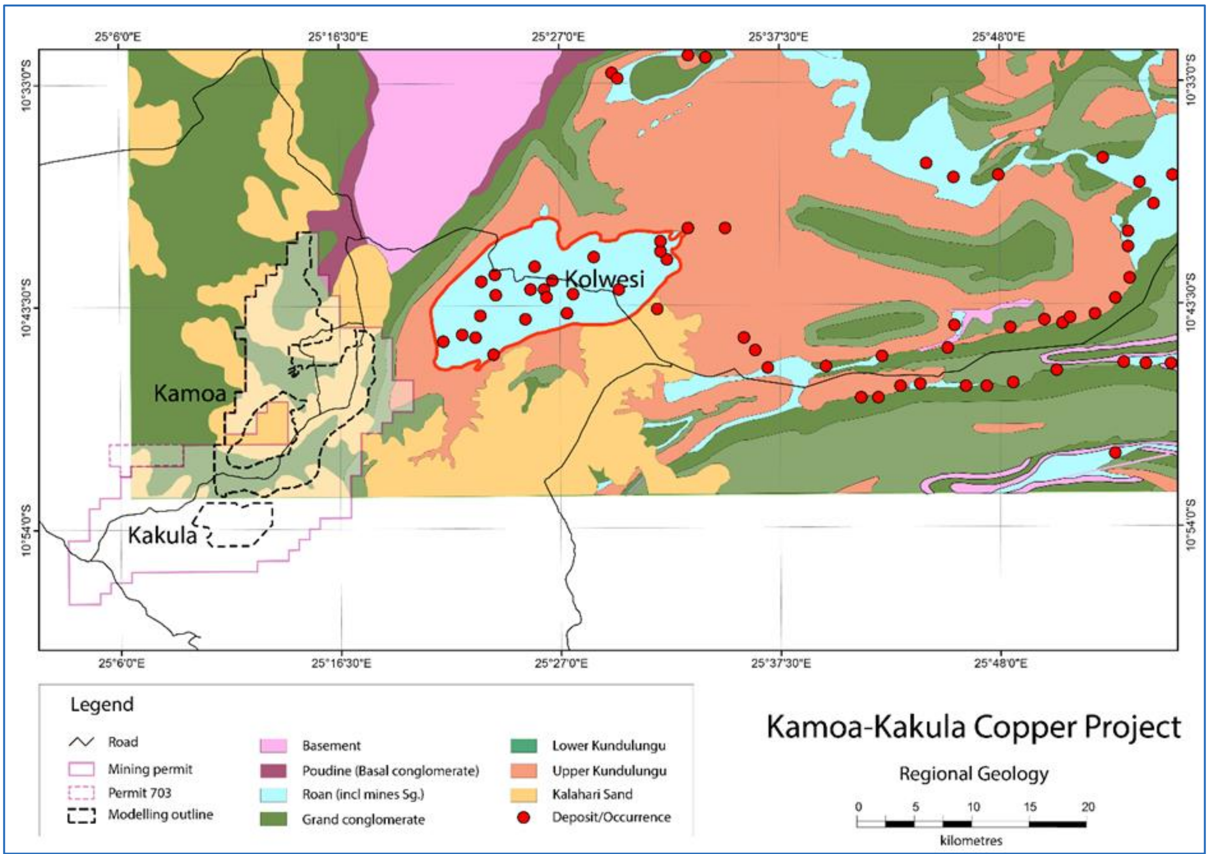


Figure provided by Ivanhoe, 2016.

7.2 Project Geology

The majority of the Project area lies on a broad, gentle plateau between two major north-north-east trending structures. To the east, and identified primarily by airborne magnetics, is the Kansoko Trend which is the interpreted boundary with the External Fold and Thrust Belt. The geology of the Kansoko Trend is currently poorly defined. To the west is a prominent escarpment and magnetic feature named the West Scarp Fault.

Between these structures a series of gentle domes occur, where the Grand Conglomerat is eroded, and the underlying Roan sandstones are exposed. The outline of the domes used in the resource model are expanded to include portions of the Grand Conglomerat that have been leached of mineralisation.

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

For reference to different areas within the Kamoa deposit, the Project area was divided into 13 prospect areas that are referred to throughout this Report (refer to Figure 7.4). In the succeeding presentation in Sections 7.0 to 14.0, two forms of copper are recognised: total copper is designated as TCu; sulphuric acid soluble copper is designated as ASCu.

Figure 7.4 Prospect Areas Within the Combined Exploitation Permits

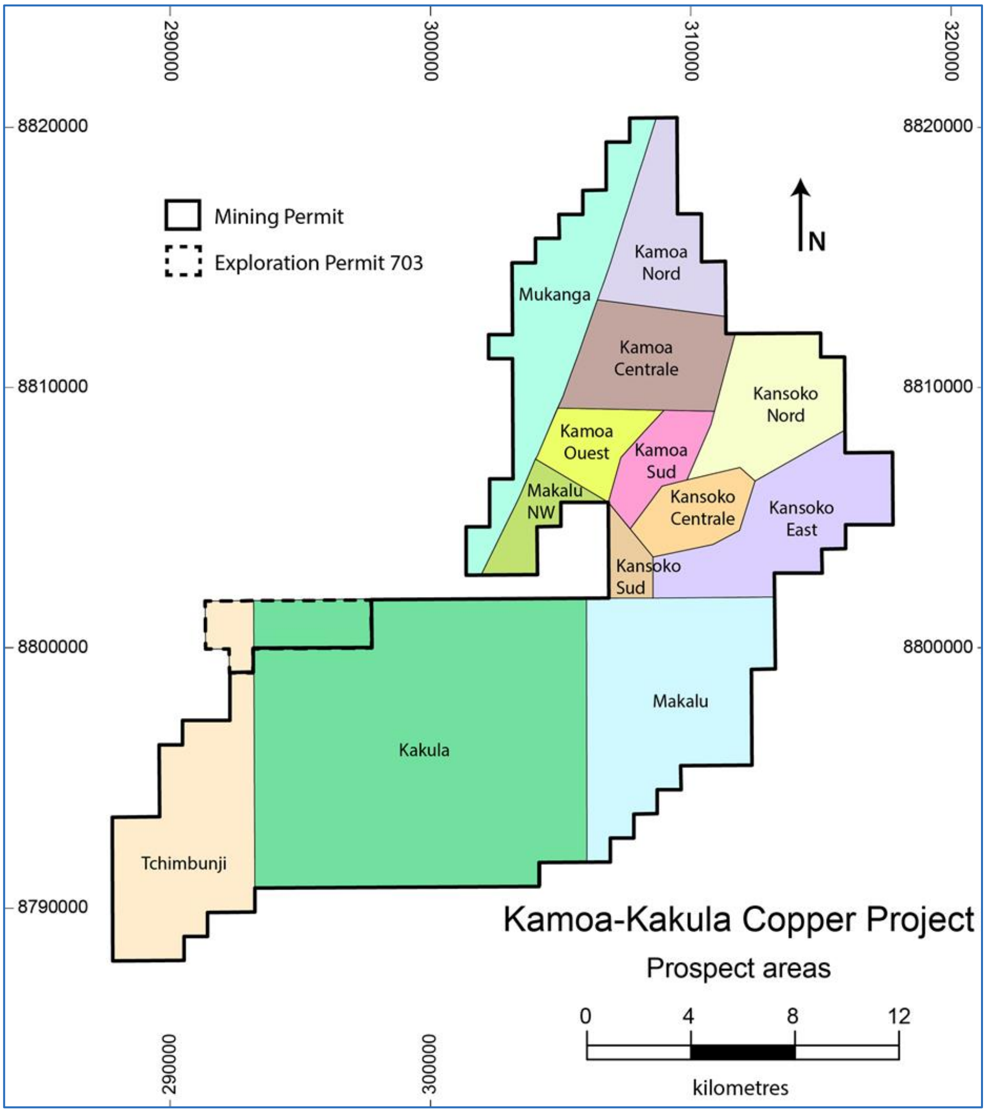


Figure provided by Ivanhoe, 2016.

7.3 Deposit Description

7.3.1 Stratigraphic Sequence

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

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

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

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

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

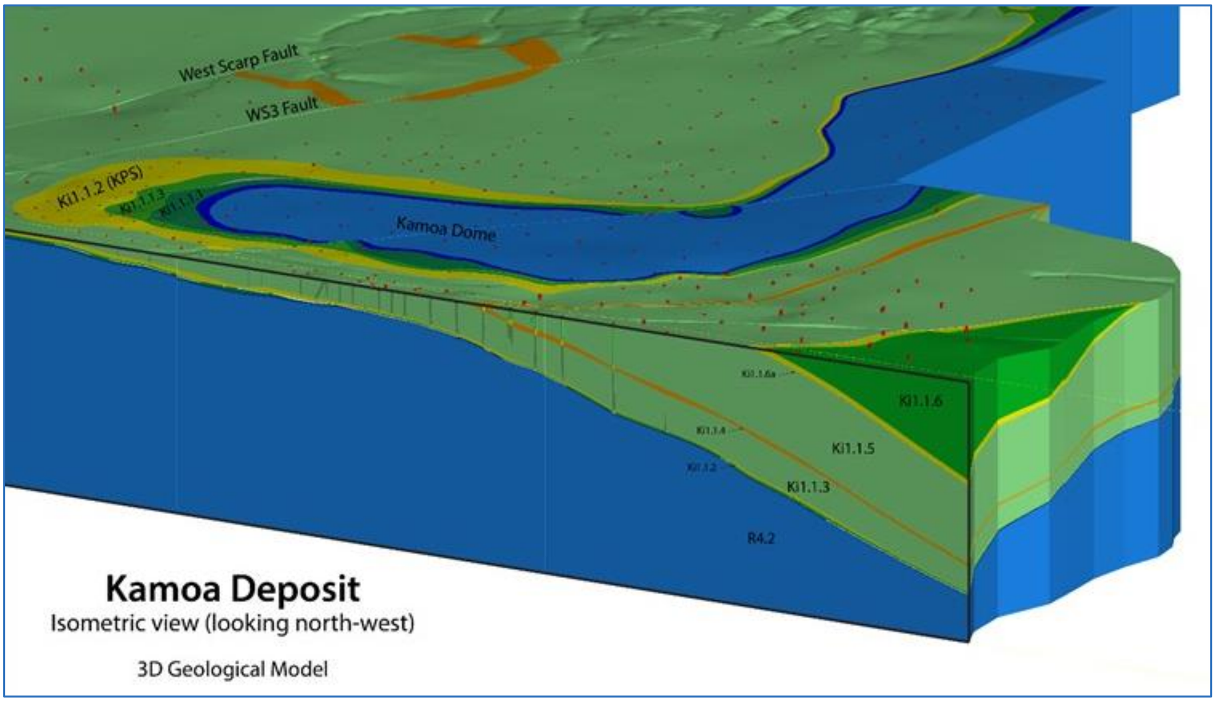


Figure provided by Ivanhoe, 2017. No scale shown given the isometric nature of the image.

As with Kamoa, sandstones of the Mwashya Subgroup of the Roan Group (R4.2) form the basal unit at Kakula. At Kamoa, a distinction is drawn between a basal clast-rich diamictite (Ki1.1.1.1) and an upper clast-poor diamictite (Ki1.1.1.3), with a variably developed intermediate siltstone (Ki1.1.1.2) sometimes separating them. The distinction of two diamictites at Kakula is not as clear. The diamictites of the Ki1.1.1 are generally clast poor and are typically more silty, suggesting that Kakula represents a more distal depositional environment relative to Kamoa. At Kakula, numerous siltstones are developed within the Ki1.1.1, especially in the lower half of the unit. These siltstones appear to be broadly continuous (Figure 7.9); however, there is no clear correlation between any specific siltstone at Kakula and the intermediate siltstone (Ki1.1.1.2) recognised at Kamoa. A key lithological unit recognised at Kakula is a laterally-continuous basal siltstone, developed just above the R4.2 contact. The basal siltstone is separated from the R4.2 contact by a narrow (often <1 m thick), yet persistently developed, clast-rich diamictite.

For modelling purposes, the identification of minor stratigraphic units within the Ki1.1.1 has not yet been attempted for the Kakula model.

Figure 7.6 Local Stratigraphy for the Kamoa Deposit

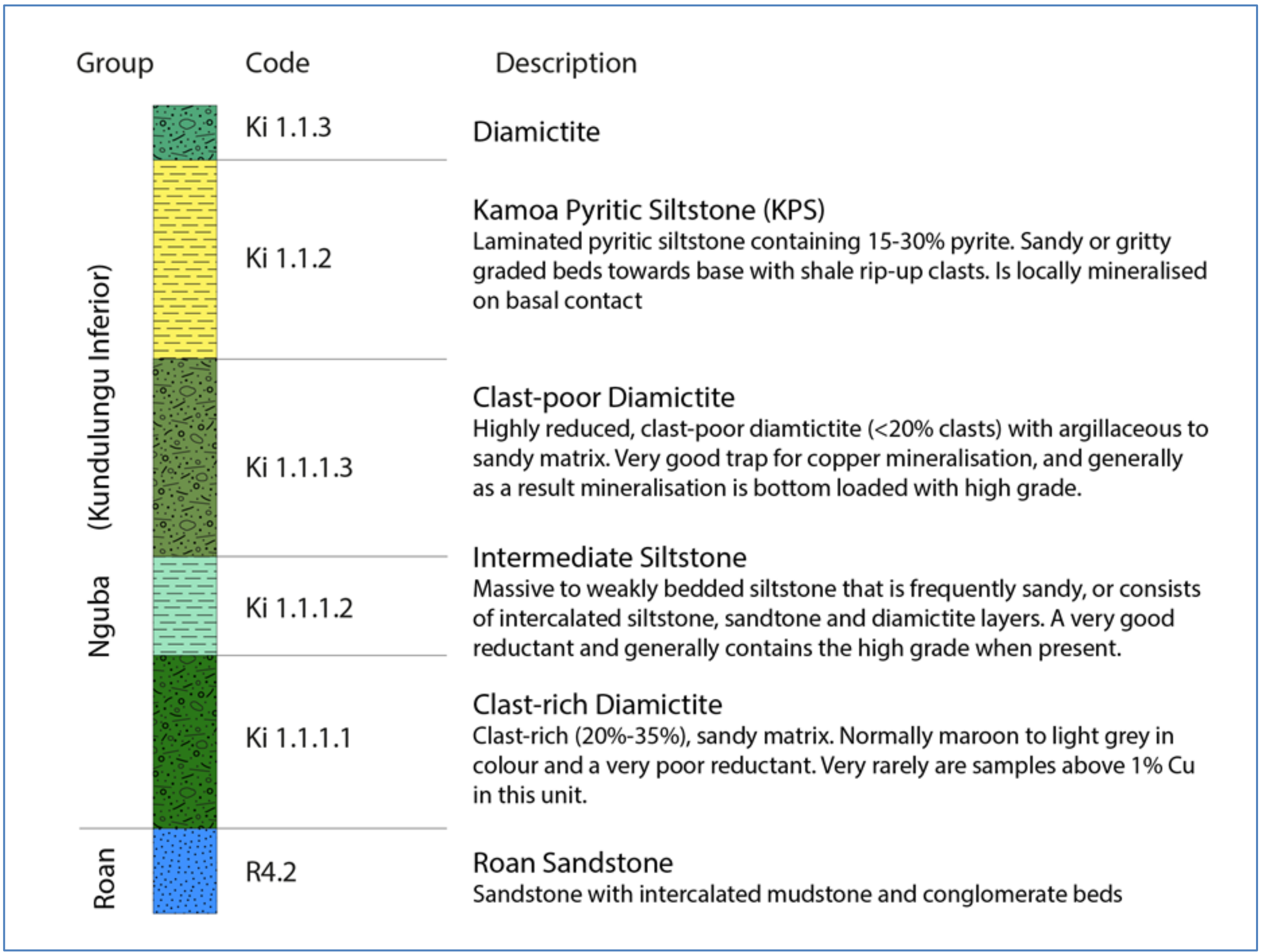
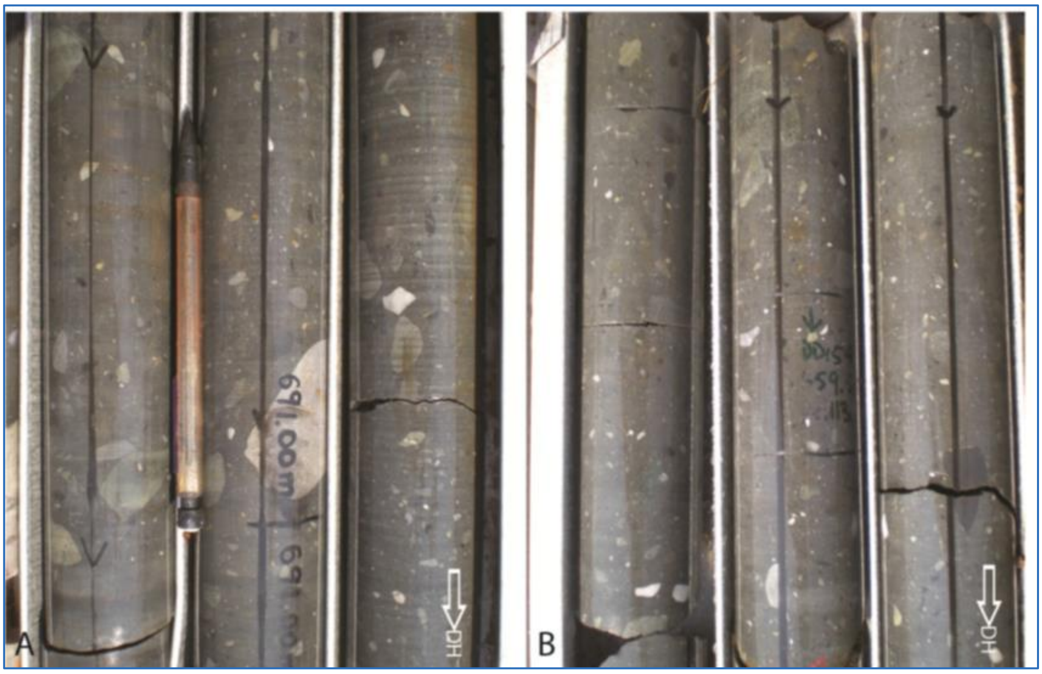


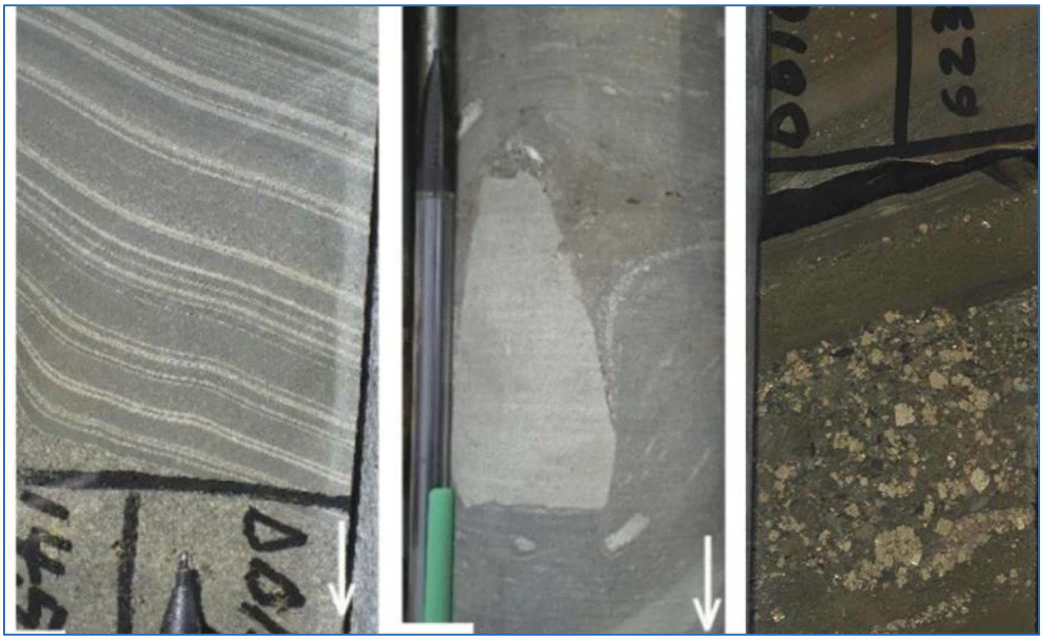
Figure provided by Ivanhoe, 2016.

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



Source: Schmandt et al (2013).

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



Source: Schmandt et al (2013).

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

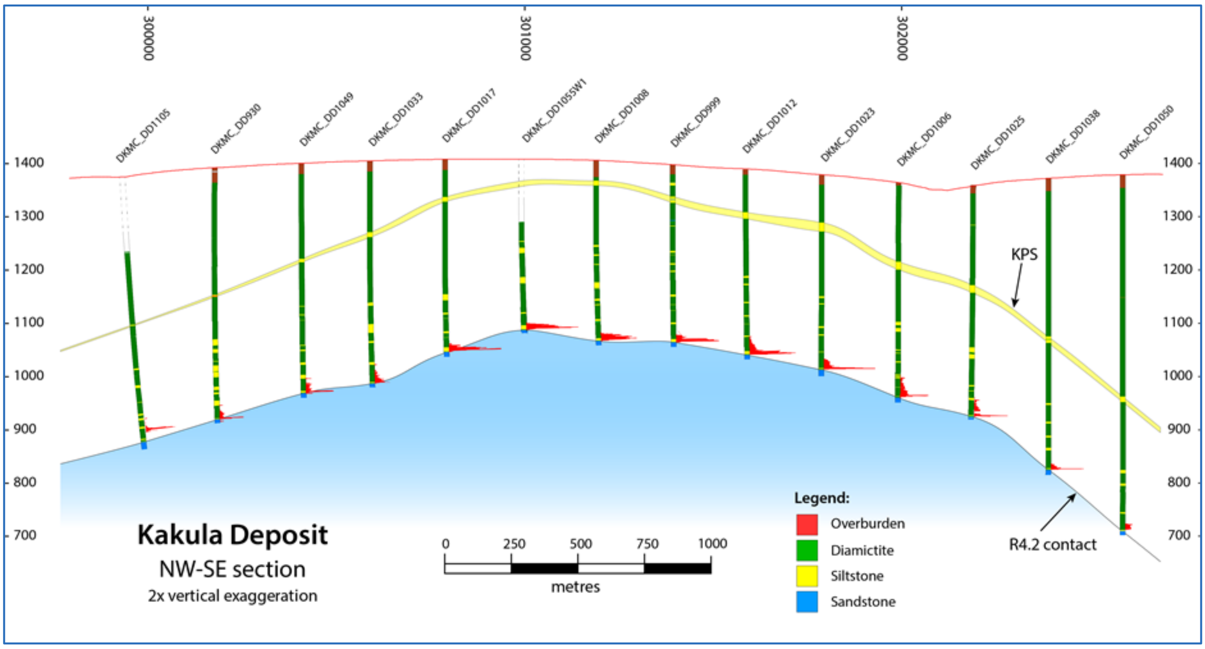


Figure provided by Ivanhoe, 2017. Red bars indicate assay intervals grading ≥ 0.5% Cu.

7.3.2 Thicknesses of Diamictite Units (Ki 1.1.1)

Vertical thickness plots for the two diamictite units and the KPS, (refer to Figure 7.10 through Figure 7.12), indicate a north-west trend to stratigraphic thickness developed across the Project area. This is particularly evident in Kansoko Sud, where changes in thickness of the Ki1.1.1.1 and Ki1.1.1.3 are evident. The thickening is very obvious on a section line perpendicular to the thickening orientation, refer to Figure 7.13.

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

In the south-west, the thickening of the diamictite units is also marked by the development of thicker siltstone sandstone siltstone units, or the development of numerous siltstone units. In this area, to aid modelling and grade estimation, the Ki1.1.1.2 has been subdivided into three units (Figure 7.16). It appears a fault (or series of faults) orientated north-west were active during sedimentation, controlling the change in thickness of the units even if the individual units have not been offset across these structures.

Figure 7.10 Ki1.1.1.1 Vertical Thickness

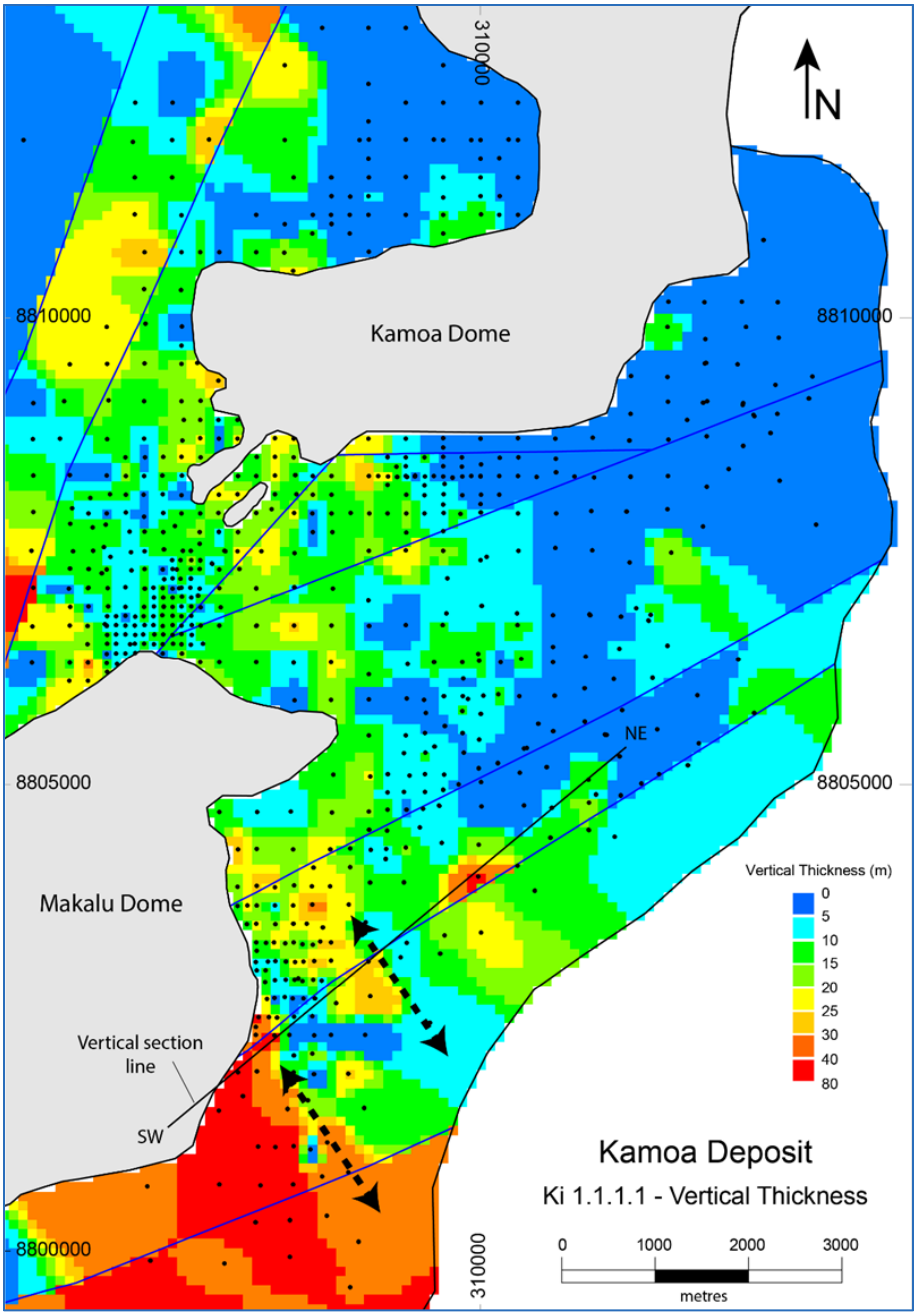


Figure provided by Ivanhoe, 2016; black line is the trace of the cross-section shown in Figure 7.13; grey areas represent domes or leached zones; blue lines are interpreted faults; black dashed lines represent north-west oriented thickness trend.

Figure 7.11 Ki 1.1.1.3 Vertical Thickness

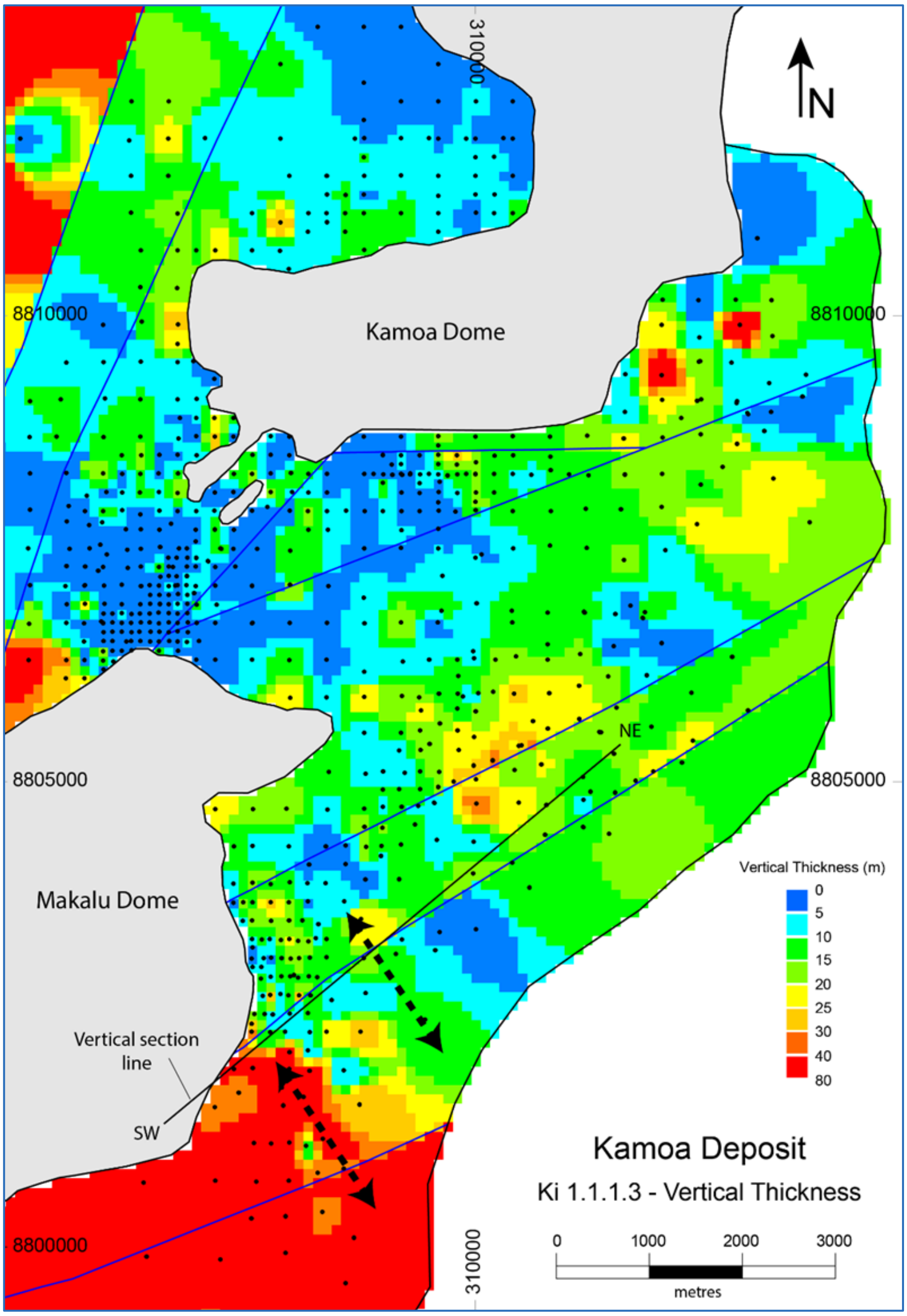


Figure provided by Ivanhoe, 2016; black line is the trace of the cross-section shown in Figure 7.13; grey areas represent domes or leached zones; blue lines are interpreted faults; dashed black lines represent north-west oriented thickness trend.

Figure 7.12 Ki 1.1.2 Vertical Thickness (KPS)

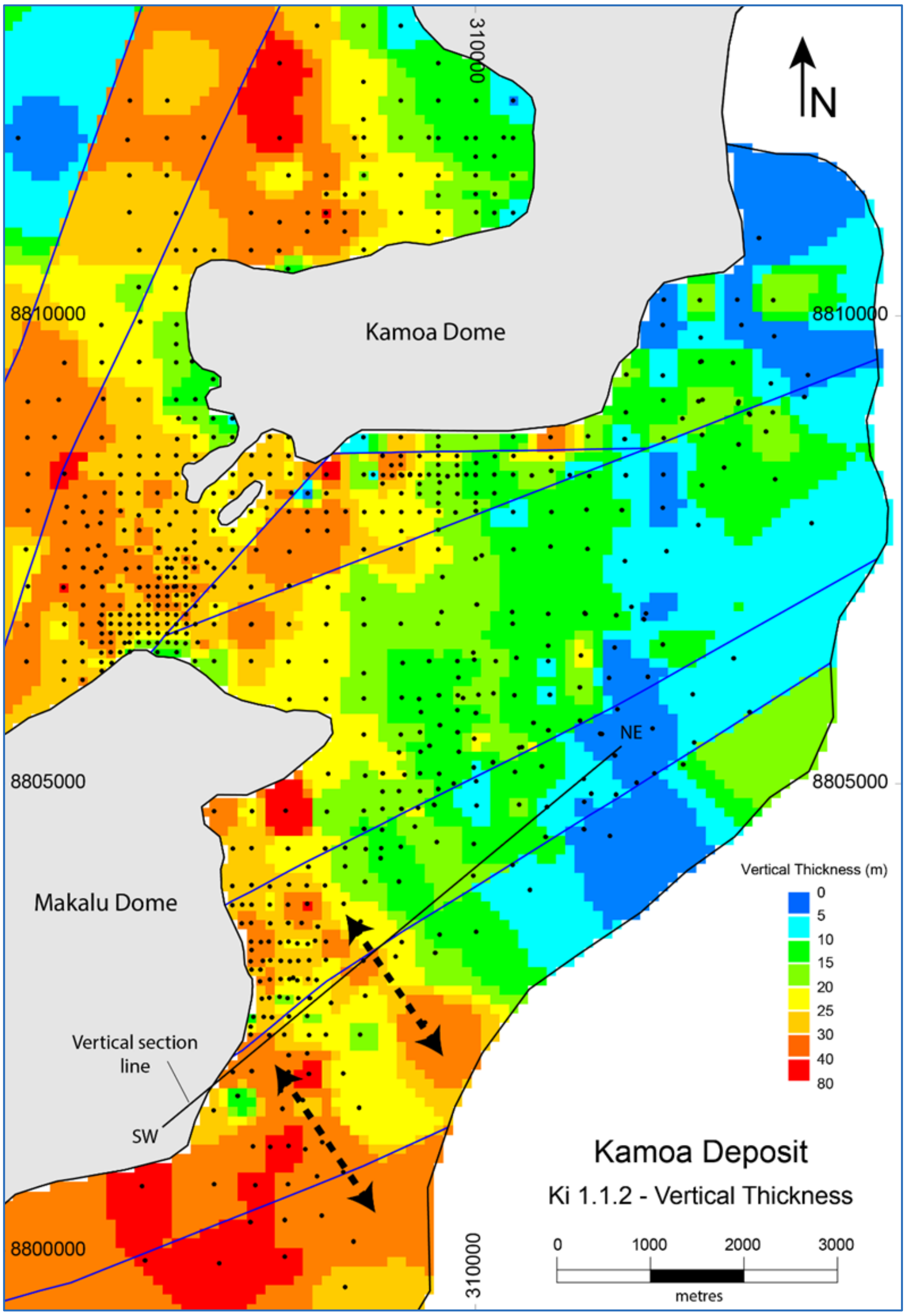


Figure provided by Ivanhoe, 2016; black line is the trace of the cross-section shown in Figure 7.13; grey areas represent domes or leached zones; blue lines are interpreted faults; dashed black lines represent north-west oriented thickness trend.

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

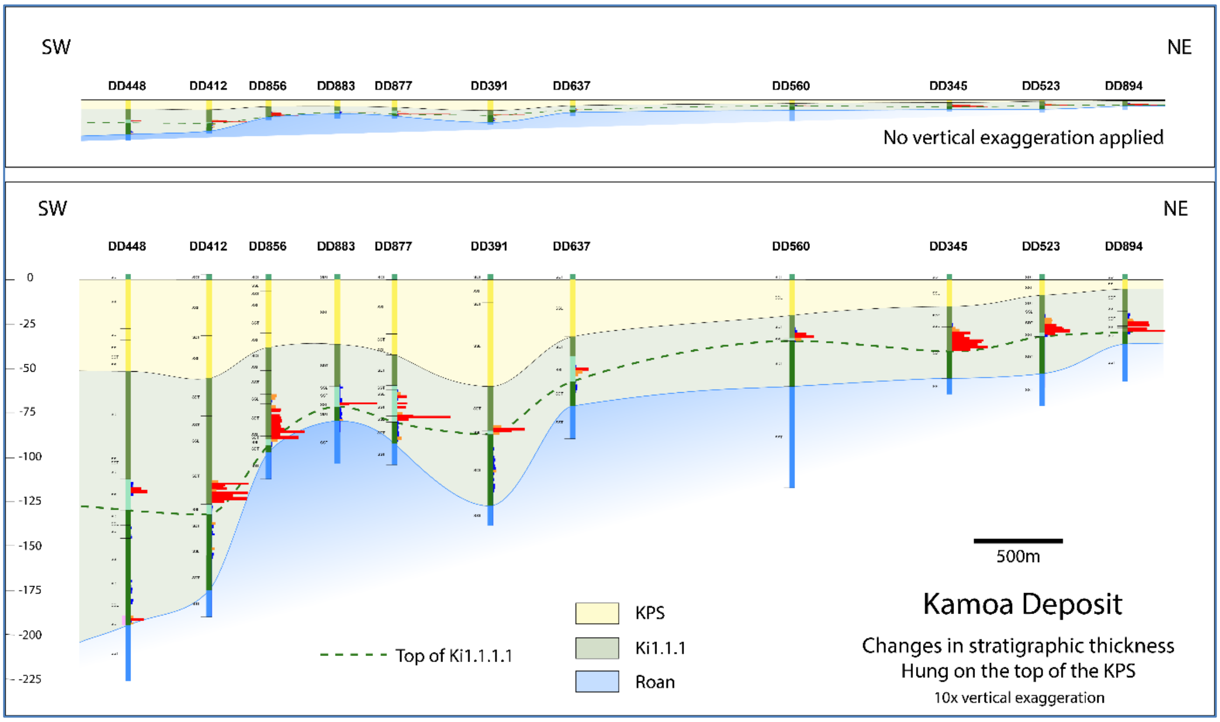


Figure provided by Ivanhoe, 2014, illustrating the thickening of units to the south-west; section line location is indicated in Figure 7.10 to Figure 7.12. Copper grades are shown as histograms with red being over 1% TCu.

Figure 7.14 Ki 1.1.1.2 Vertical Thickness

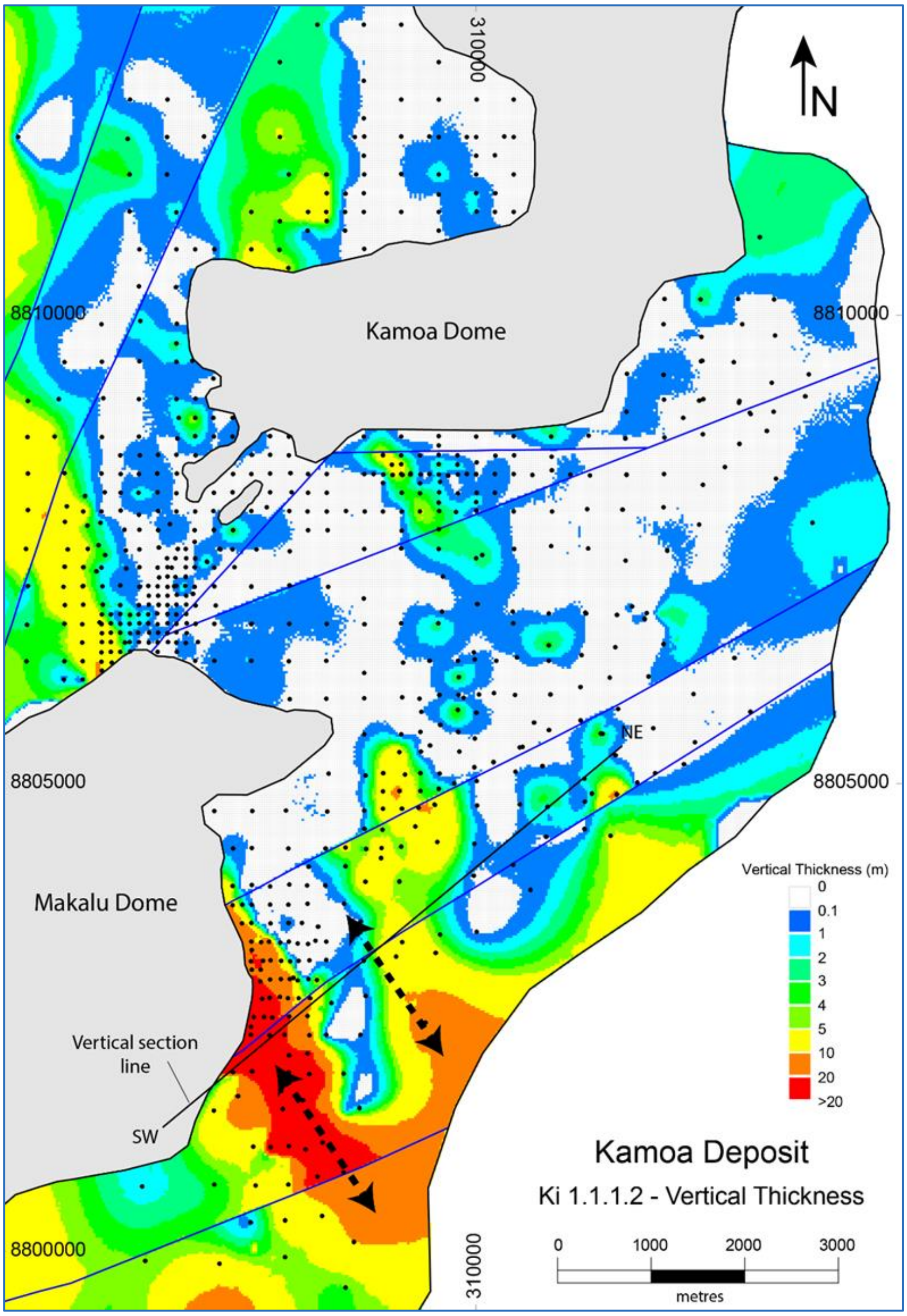


Figure provided by Ivanhoe, 2016; black line is cross-section shown in Figure 7.13; grey areas represent domes or leached zones; blue lines are interpreted faults; dashed black lines represent north-west oriented thickness trend.

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

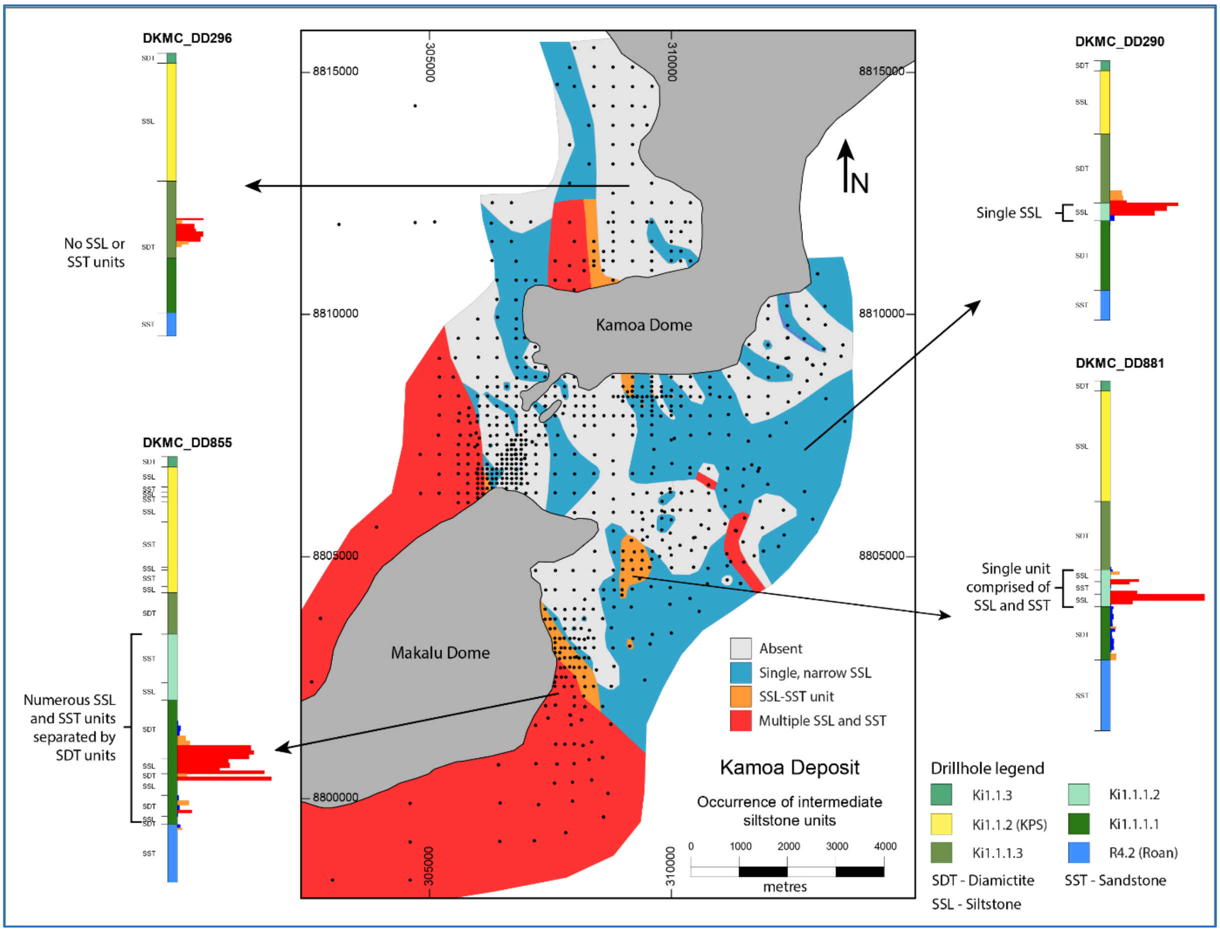


Figure Provided by Ivanhoe, 2018. Copper grades, shown as red histograms (where over 1% TCu).

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

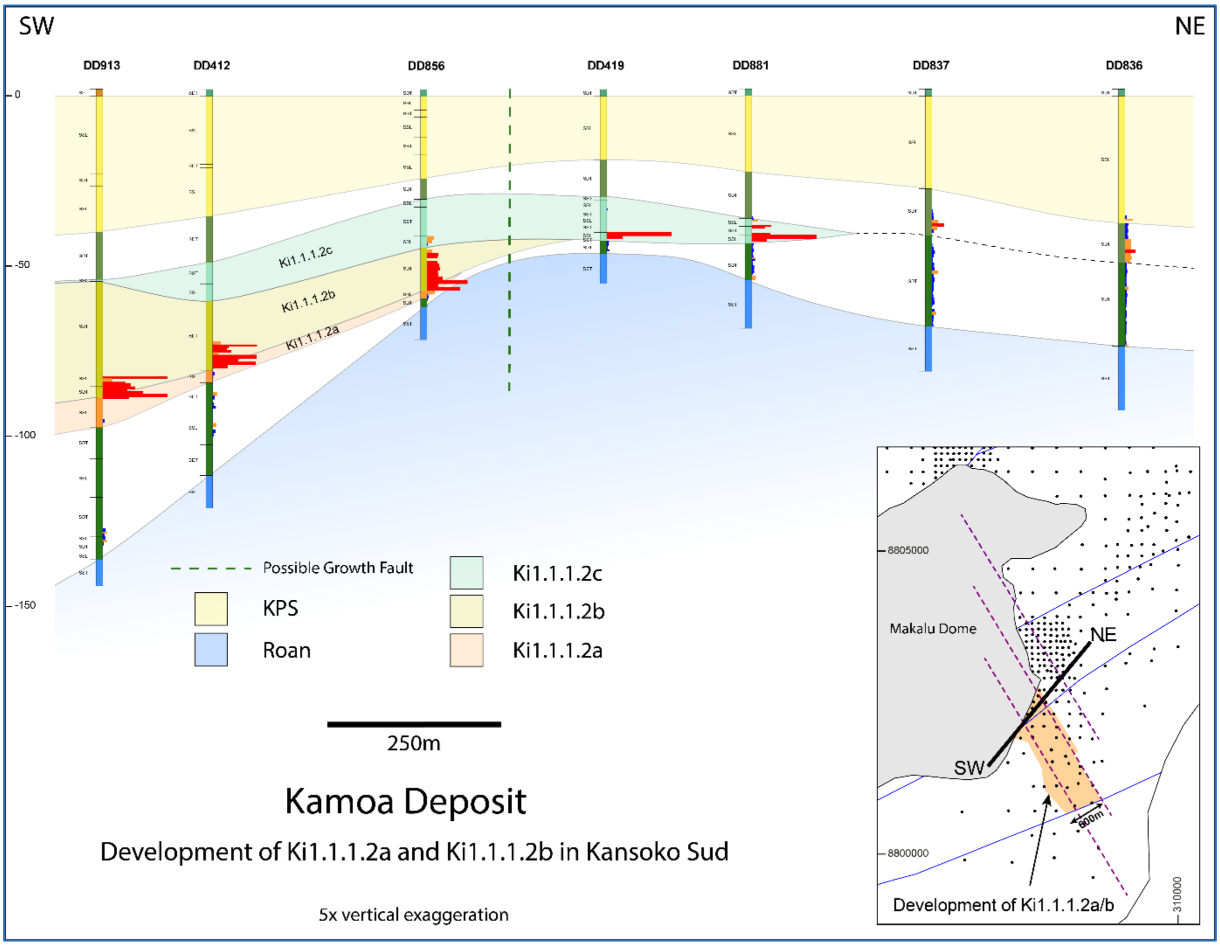


Figure provided by Ivanhoe, 2017. Copper grades in percent, shown as red histograms if over 1% TCu.

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

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

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

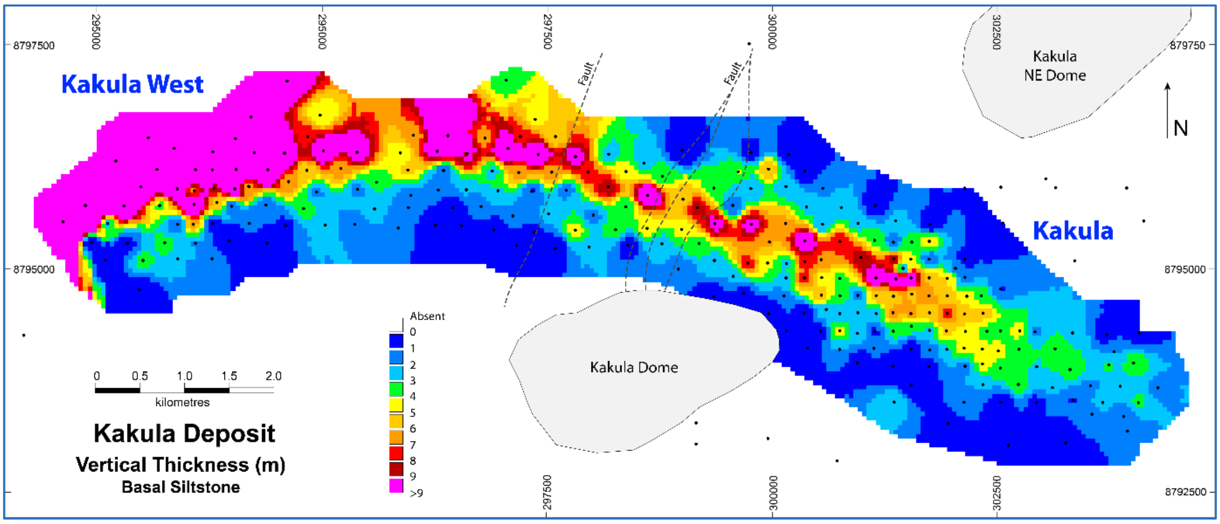


Figure provided by Ivanhoe, 2018. Vertical thickness estimated using an isotropic search.

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

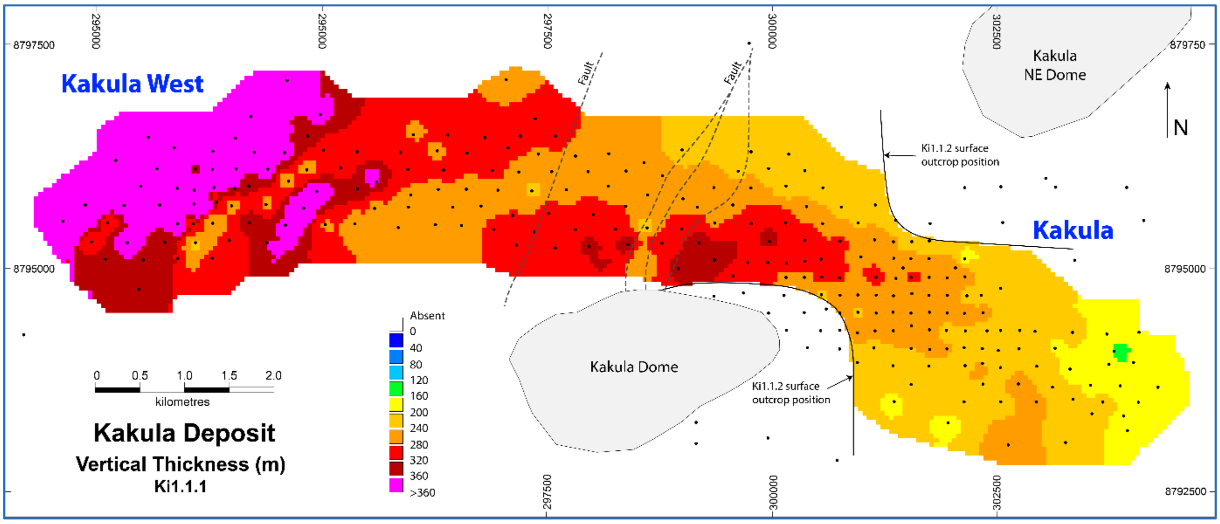


Figure provided by Ivanhoe, 2018. Vertical thickness estimated using an isotropic search.

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

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

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

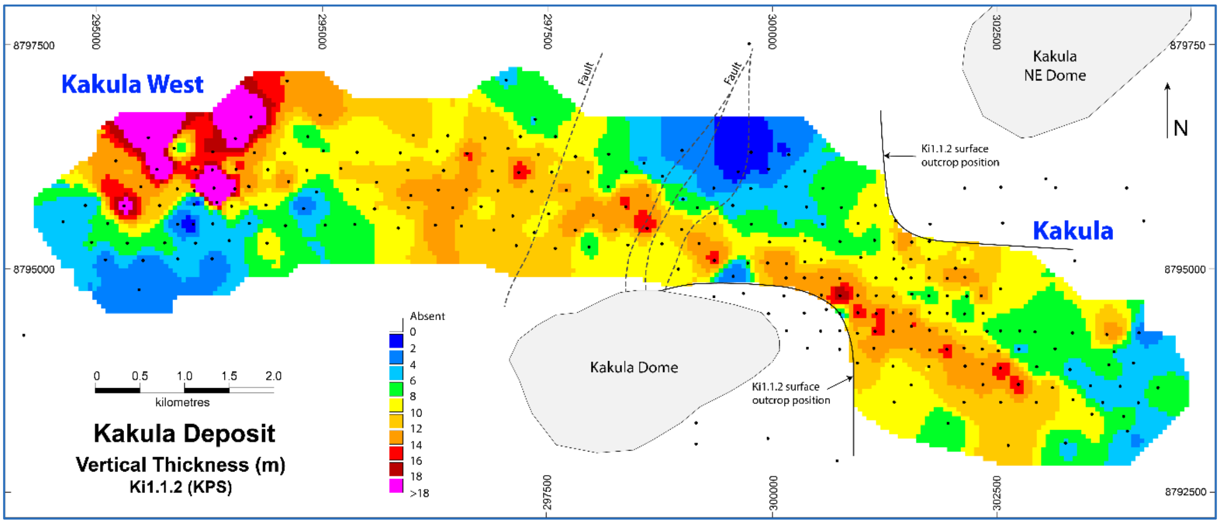


Figure provided by Ivanhoe, 2018. Vertical thickness estimated using an isotropic search.

7.3.3 Igneous Rocks

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

7.3.4 Structure

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

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

Figure 7.20 Structural Model Overlaid on Second Vertical Derivative Magnetic Image

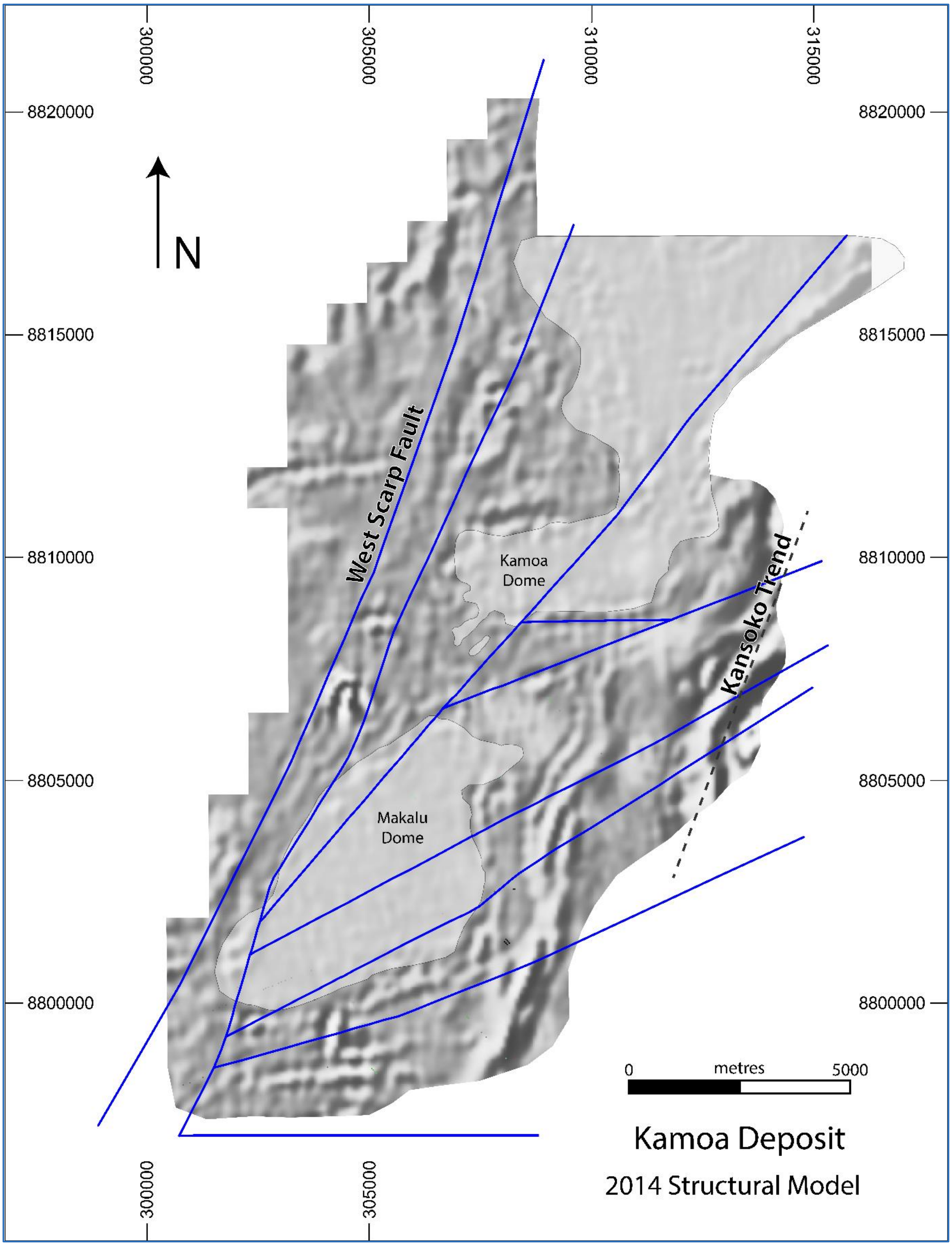


Figure provided by Ivanhoe, 2016; blue lines are interpreted fault traces; light grey areas are domes and surrounding leached zones.

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

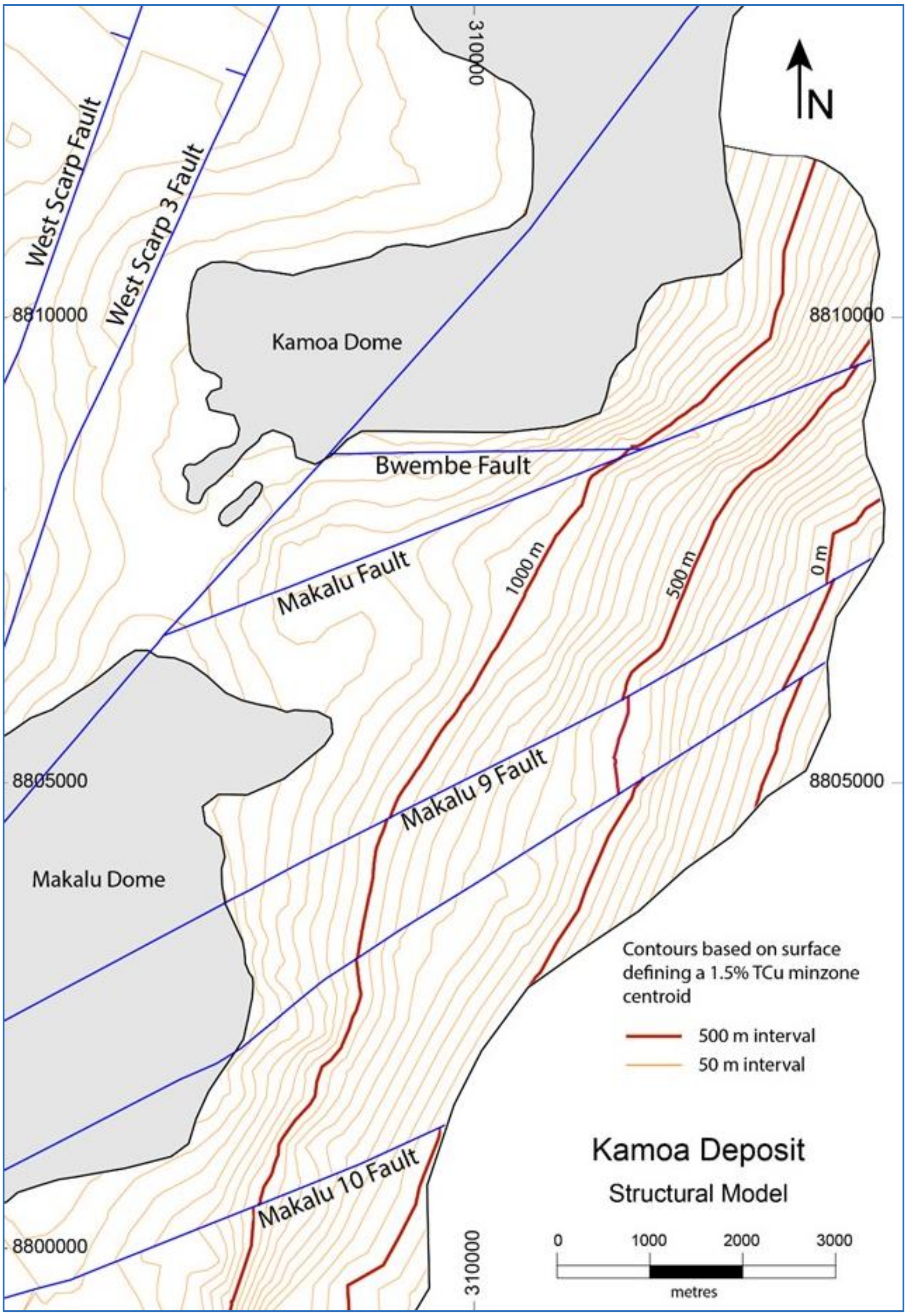


Figure provided by Ivanhoe, 2018.

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

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

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

Microstructures are commonly observed in core, particularly in the finely laminated siltstone units. Observed offsets are on the millimetre or centimetre scale, with either normal or reverse sense of movements identified. A steep to vertical foliation is defined by the alignment of clasts or minerals within the matrix, and the alignment of fine and coarse-grained sulphides. In rare cases, unusually steep bedding is identified to occur over intervals of 0.5 m to 2 m. These occurrences often coincide with the high copper grades (>5% TCu) and have been observed to align on the north-northwest growth fault trend evident from changes in thickness of individual stratigraphic units. In these areas, microstructures are usually flat-lying, suggesting they formed earlier and were rotated during the time the bedding was steepened. The foliation, however, remains steep regardless of the nature of the bedding, suggesting it is a later overprint (Twite, 2016). Examples of these features are shown in Figure 7.23.

Figure 7.22 Structural Influences on Topography

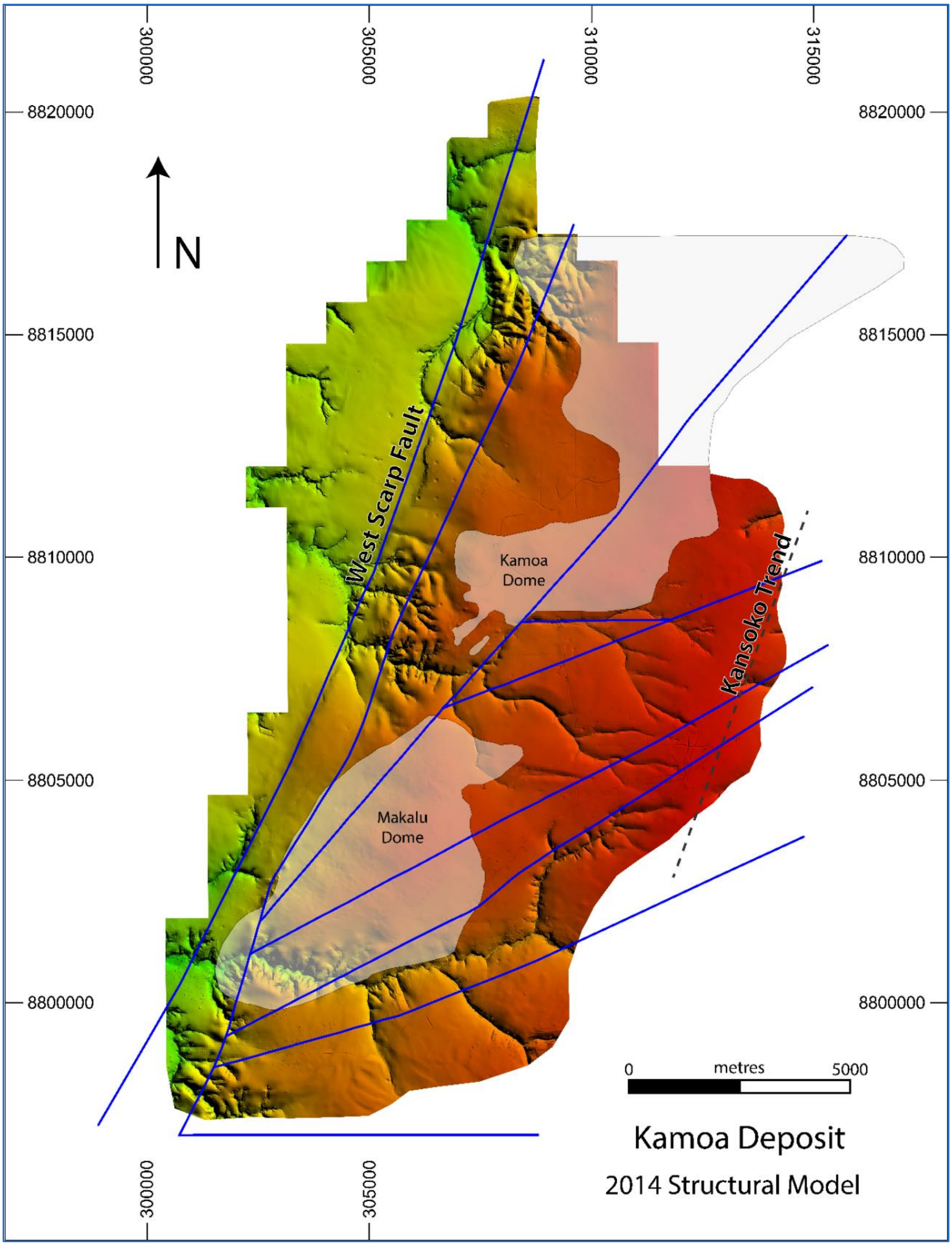


Figure provided by Ivanhoe, 2016.

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

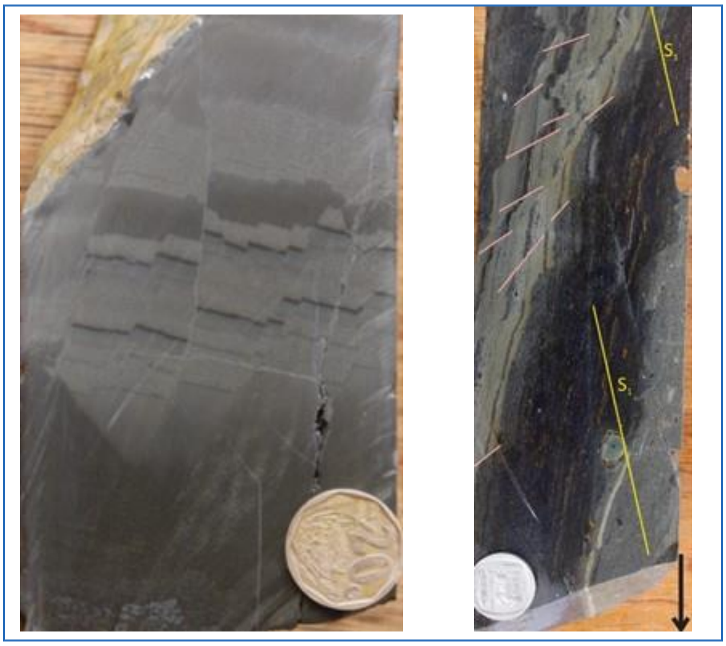


Figure provided by Ivanhoe, 2016. Yellow line indicates foliation orientation. Whitish-pink lines identify microstructures offsetting layering.

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

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

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

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

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

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

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

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

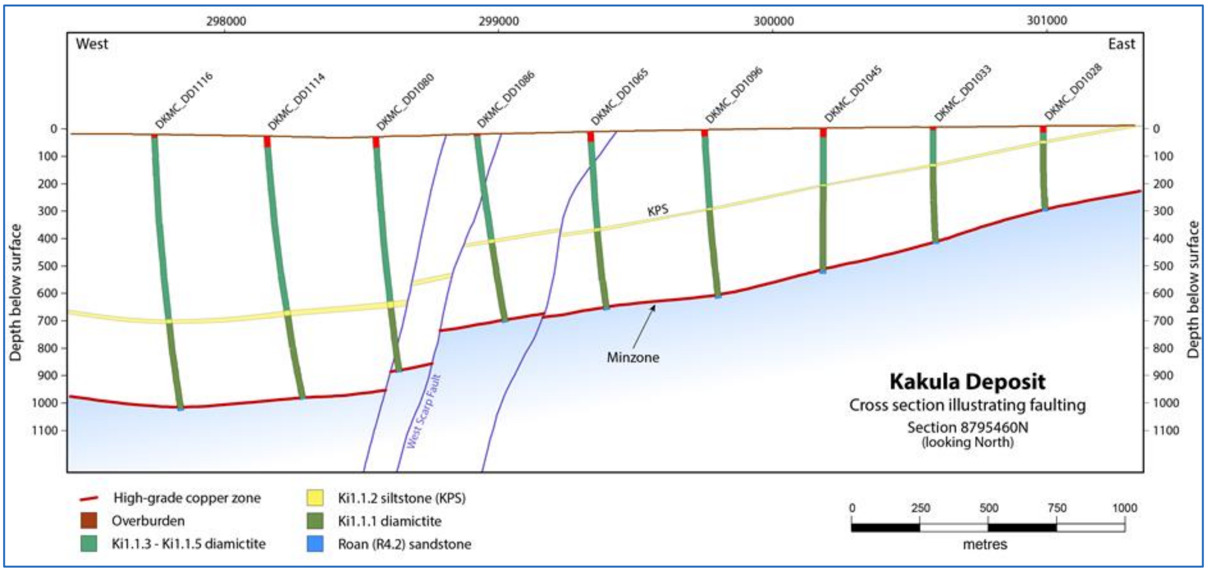


Figure provided by Ivanhoe, 2018.

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

Figure provided by Ivanhoe, 2016.

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

Figure provided by Ivanhoe, 2016.

The structural model used in construction of the Kakula Mineral Resource estimate includes the four north-north-east structures. The four modelled faults cause the Mineral Resource area to be divided into five structural blocks, where offsets in the elevation of the centroid of the 3% mineralised zone are modelled (Figure 7.27). Drilling has confirmed that the edges of the Kakula and Kakula West domes are characterised by areas of pronounced steeper dip resulting either from syn-deposition faulting or uplift during inversion. Additional drilling is planned in these areas to confirm the overall geometry.

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

The shallowest portion of the Kakula deposit between the two domes (Figure 7.28) is also gently-dipping. The average dip of the mineralised zone in the Indicated Mineral Resource area is 17°, while the average dip is 19° in the Inferred Mineral Resource area.

Figure 7.27 Structure Model for the Kakula Resource Area showing contours for the centroid of the 3% mineralised zone (SMZ30)

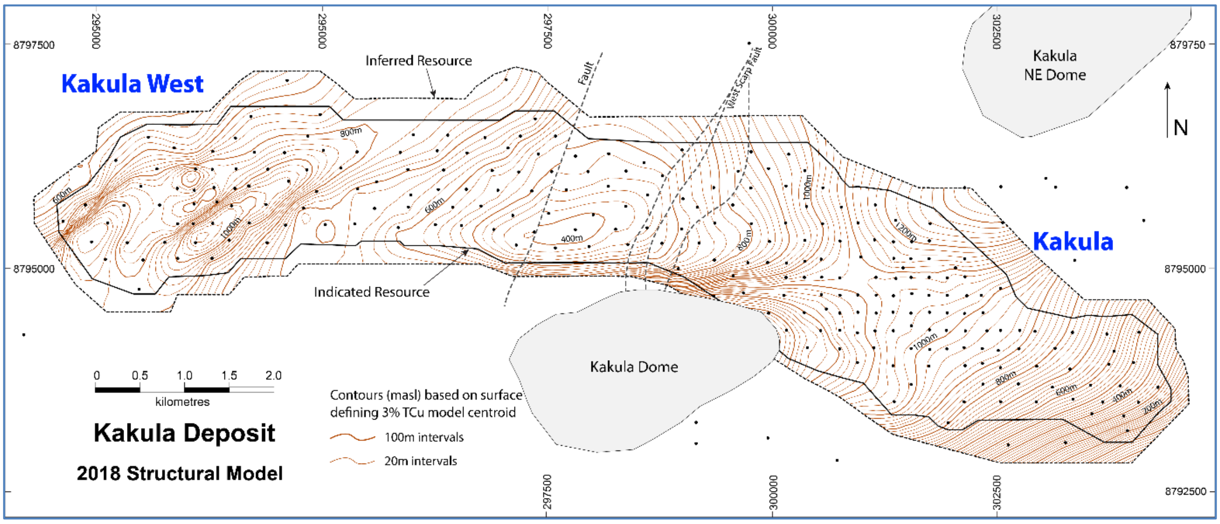


Figure provided by Ivanhoe, 2018.

Figure 7.28 Depth Below Surface for the Kakula 3% Copper Grade Shell

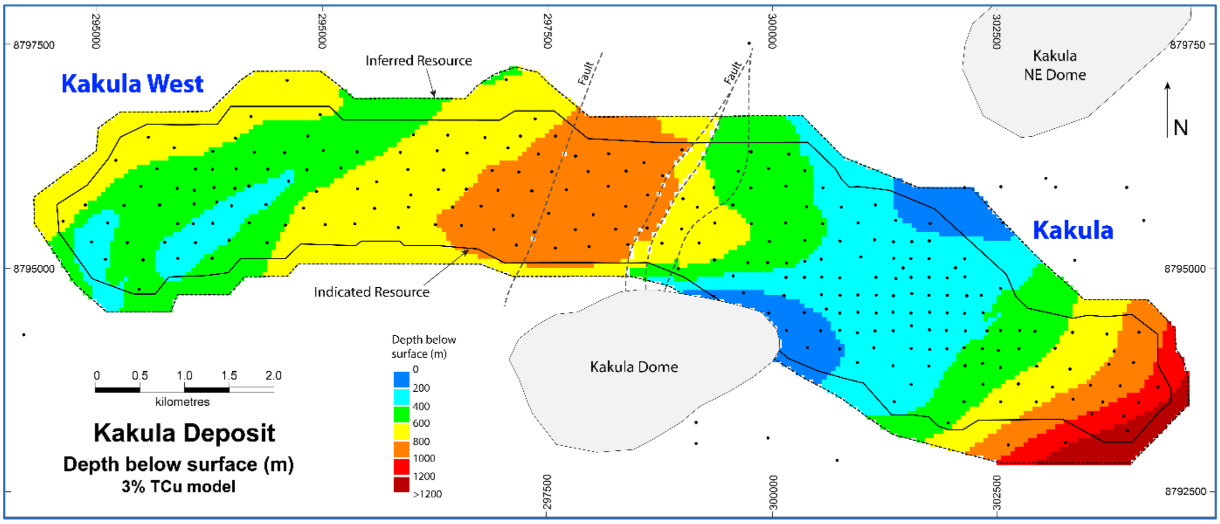


Figure provided by Ivanhoe, 2018. Missing blocks are due to non-vertical fault offsets.

7.3.5 Metamorphism

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

7.3.6 Alteration

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

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

7.3.7 Mineralisation

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

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

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

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

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

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

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

Mineral Zonation

Two broad categories of lateral zonation are evident at Kamoa (hypogene and supergene); however, within the hypogene, additional lateral zonation is evident based on the relative abundance of chalcopyrite, bornite and chalcocite. The dominant sulphide species within the mineralised zone is interpreted to be a lateral mineral zonation. The change from supergene to hypogene is generally transitional with a strongly developed vertical zonation evident in the hypogene (refer to Figure 7.29). There is significant pyrite mineralization above the mineralized horizon that could possibly be exploited to produce pyrite concentrates for sulphuric acid production.

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

Figure 7.29 Schematic of Mineral Zonation at Kamoa

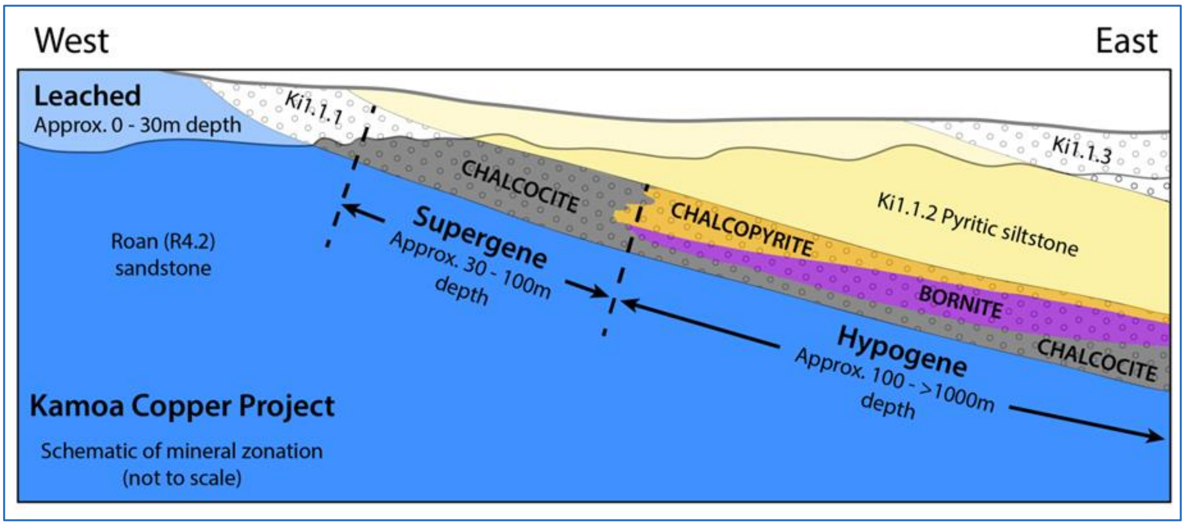


Figure is schematic and not to scale. Leached zone ranges from 0 to 30 m vertical depth from surface. Supergene zone ranges from 30 m to 100 m vertical depth from surface. Hypogene typically extends from 100 m vertical depth to more than 1,560 m. The supergene/hypogene interface is typically at less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

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

At Kamoa, chalcopyrite dominates, primarily as fine-grained disseminations in the diamictite matrix, although very coarse chalcopyrite can form as elongated grains up to 5 mm in length rimming clasts or fragments, or defining strain shadows to clasts (Figure 7.30). A steep to vertical foliation is defined by the weak alignment of clasts and minerals within the matrix, but is often best displayed by the alignment of fine and coarse-grained sulphides (Twite, 2016) (Figure 7.30). Bornite is typically fine-grained and disseminated in the matrix of the diamictite. When well developed, the fine-grained bornite is recognised visually through a significant darkening of the diamictite matrix. Chalcocite almost always occurs as fine-grained disseminations, particularly within the intermediate siltstone (Ki1.1.1.2). Supergene zones, in close proximity to dome edges, are typically chalcocite-dominant.

Figure 7.30 Strain-Shadow in DKMC_DD909

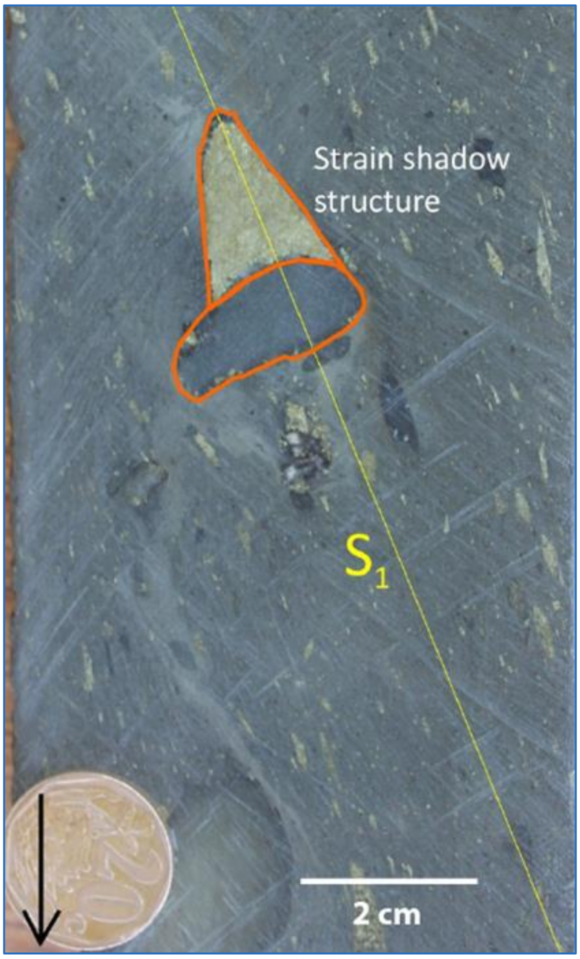


Figure provided by Ivanhoe, 2016. S_1 defines the orientation of the steep foliation.

Relative to Kamoa, the Kakula deposit is very different in its style of mineralisation. Whilst the vertical hypogene zonation is still evident at Kakula, the chalcopyrite and bornite zones are very narrow, with a very gradual transition downward from bornite to chalcocite, followed by a zone (typically within the basal siltstone) that is chalcocite-dominant (Figure 7.35). Whilst still dominantly fine-grained, numerous examples of coarse to massive chalcocite are evident in the highest-grade intersections (Figure 7.31).

Figure 7.31 Examples of Coarse to Massive Chalcocite at Kakula

DKMC_DD1055 at 347_m

DKMC_DD996 at 376_m

Figures provided by Ivanhoe, 2016.

Based on molar mass ratios, the theoretical TCu: S ratio (total copper divided by sulphur) for chalcopyrite, bornite and chalcocite was calculated and used to guide the identification of a change in sulphide species within individual drillholes. An overlap between sulphide species is generally observed in core and in thin sections; however, investigation of the TCu: S ratio shows that changes from one sulphide species to another can happen over distances of <1 m (Figure 7.32).

From the TCu: S ratio it is evident that the zonation develops regardless of the changes in copper grade. This is evident in DKMC_DD963, where the change to a theoretical chalcocite TCu: S occurs in the Ki1.1.1.1 even though the TCu grade has dropped below 1% (Figure 7.32). The full vertical zonation is not always developed, with chalcocite often being weakly developed to absent. The use of TCu: S ratios becomes unstable when TCu grades are <0.1% or sulphur values are <0.04%.

A scatter plot of TCu and S for all samples within the mineralised zone (Figure 7.34) shows a clear alignment along the different sulphide species, with a degree of scatter between these relating to transitional zones.

In contrast to Kamoa, mineralisation at Kakula is characterised by being chalcocite dominant, with gradual transition upward to bornite zones (Figure 7.33 and Figure 7.35). Chalcopyrite is observed in the core, but typically occurs outside of the defined grade shells, except in peripheral areas at Kakula West where the overall mineralised zone has narrowed, incorporating the full zonation within the defined grade shells. The grade shells are manually selected from inspection of assays. They are referred to as selective (or selected) mineralized zones (SMZs).

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

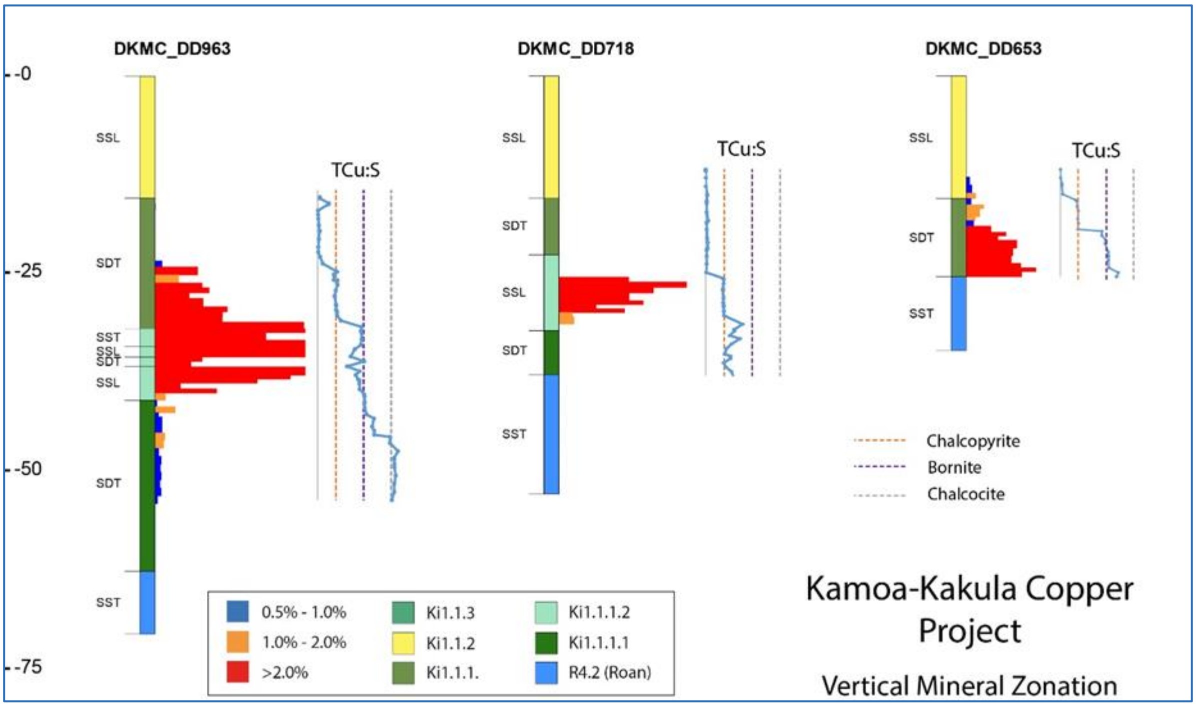


Figure provided by Ivanhoe, 2018.

Figure 7.33 Examples from Three Drillholes from Kakula of Vertical Mineral Zonation Evident Based on TCu: S Ratios

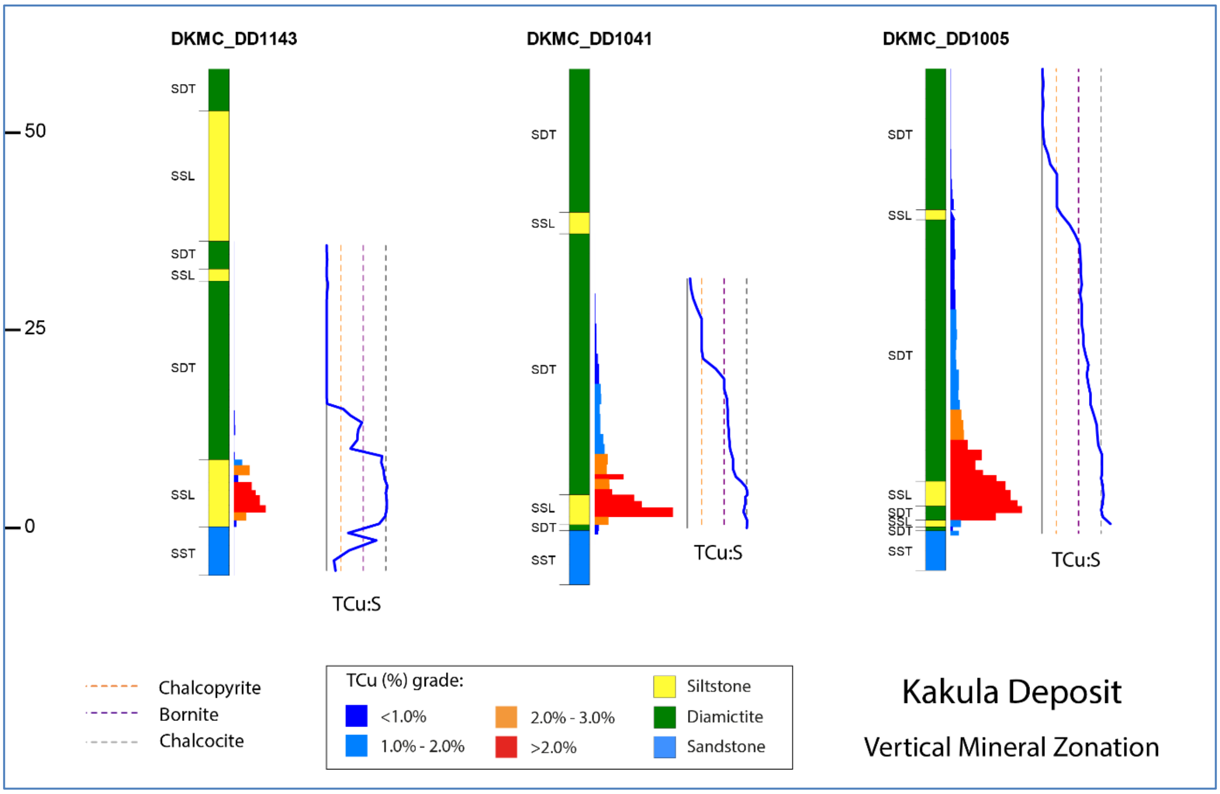


Figure provided by Ivanhoe, 2018.

Figure 7.34 Scatter Plot Illustrating Copper Sulphide Species Within the Mineralised Zone at Kamoa. Theoretical TCu: S Ratios for Chalcopyrite (orange), Bornite (purple) and Chalcocite (grey) are Based upon Molar Mass Ratios

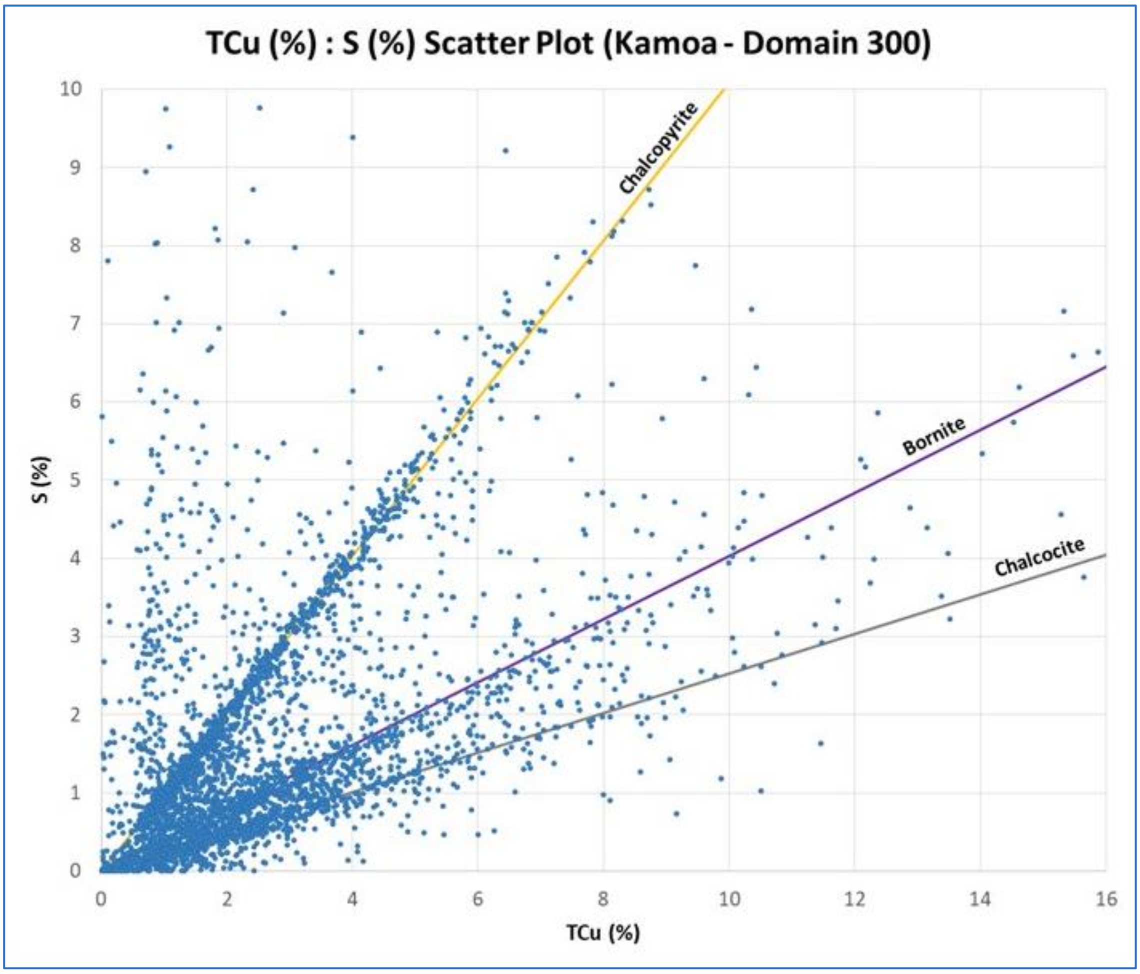


Figure provided by Ivanhoe, 2017.

Figure 7.35 Scatter Plot Illustrating Copper Sulphide Species Within the 3% Grade Shell (SMZ30) at Kakula. Theoretical TCu: S Ratios for Chalcopyrite (orange), Bornite (purple) and Chalcocite (grey) Based Upon Molar Mass Ratios

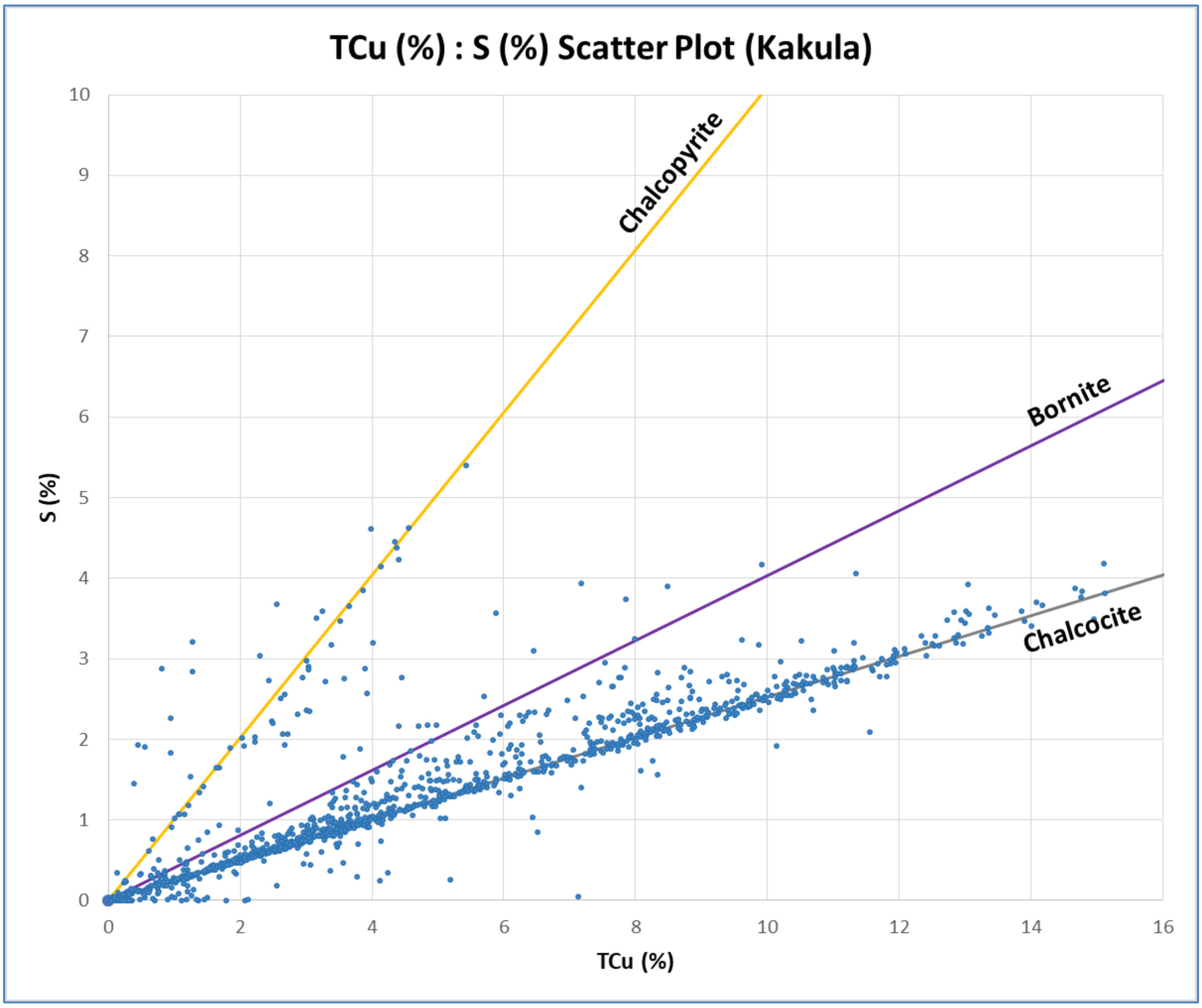


Figure provided by Ivanhoe, 2018.

Stratigraphic Position of Mineralisation

On a regional basis, the mineralisation is located at the base of the Grand Conglomerat, on (or close to) the Roan contact. Locally, using the more detailed project stratigraphy, it is evident that at Kamoa, mineralisation can be hosted in a number of different units. The mineralisation is not erratically developed in various stratigraphic positions; its position moves consistently and predictably from one unit to another, refer to Figure 7.36.

Figure 7.36 Stratigraphic Section Showing Continuity of Mineralisation Near Base of Ki 1.1.1.3 at the Kamoa Deposit (8807500N looking North)

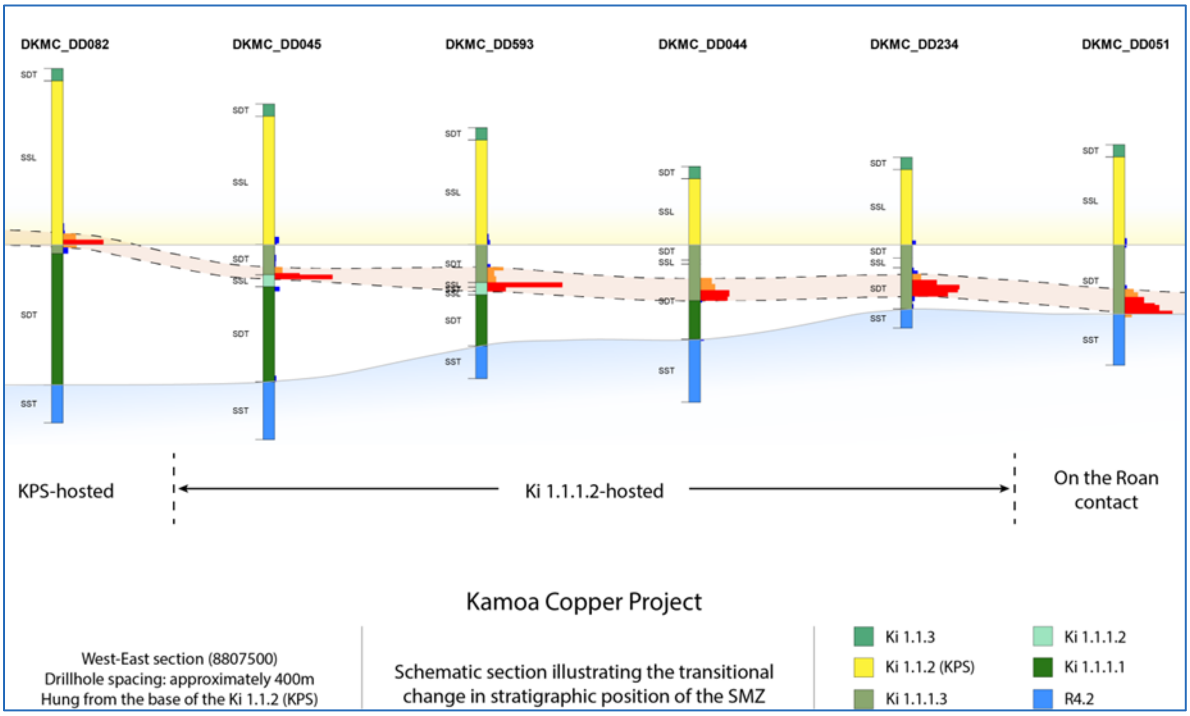


Figure provided by Ivanhoe, 2014. Copper grades in percent, shown as red histograms if over 1% TCu.

The nature of the copper grade distribution is related to its stratigraphic position and the localised development of lithological units. Where the mineralisation is located on the Roan contact, the mineralised interval is thick, and has a very strongly-developed bottom-loaded profile. Where the mineralisation is located at the base of the clast-poor diamictite (Ki1.1.1.3), the profile is typically bottom-loaded (if no intermediate siltstone is developed), or complex if one or more siltstone layers are developed. In the Kansoko Sud and Makalu areas, numerous siltstone layers developed within the diamictite cause the grade profile to become bimodal or even top-loaded. Where the mineralisation is hosted at the base of the KPS, it is typically narrow (but often high-grade), with a middle-loaded profile. The stratigraphic position of the mineralisation has been identified across the Project, refer to Figure 7.37.

Figure 7.37 Facies in which Mineralisation Occurs

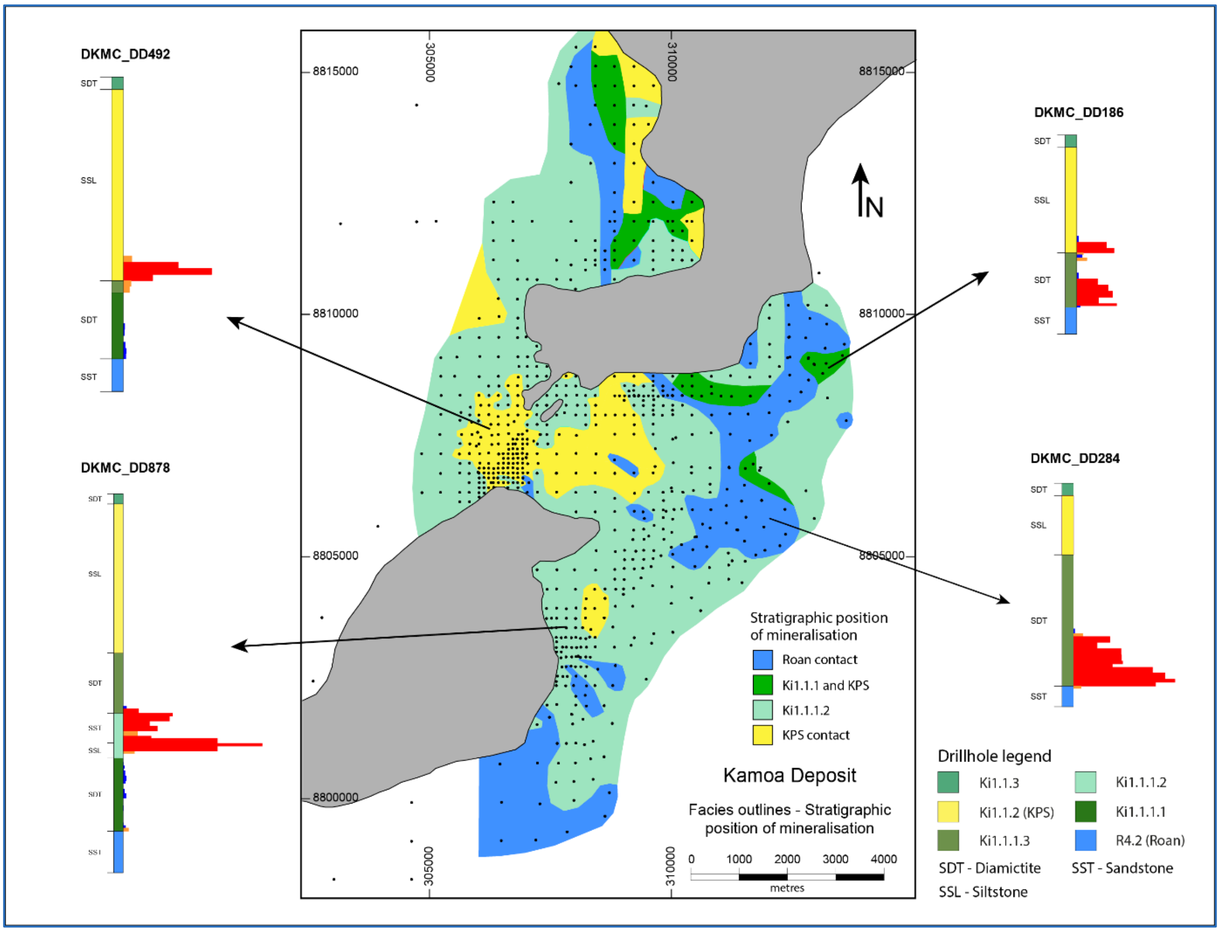


Figure provided by Ivanhoe, 2016. Copper grades in percent, shown as red histograms if over 1% TCu.

At Kakula, the mineralisation is consistently located just above (<5 m) the Roan contact. Immediately above the Roan contact, a narrow, weakly- to poorly-mineralised clast-rich diamictite is developed, above which the basal siltstone occurs. Where this siltstone is present, it forms a very sharp contact with the highest-grade intersections. Grade profiles are almost always strongly bottom-loaded, allowing for well developed lateral continuity at higher cut-offs (3% TCu) (Figure 7.38). In their shape, they resemble those at Kansoko Centrale, where the mineralisation occurs directly on the Roan contact; however, the Kakula grade profiles are usually considerably thicker and higher-grade (Figure 7.39).

Figure 7.38 Plan Image Illustrating the Continuity of High Grades due to the Bottom-Loaded Nature of the Mineralised Zone at Kakula

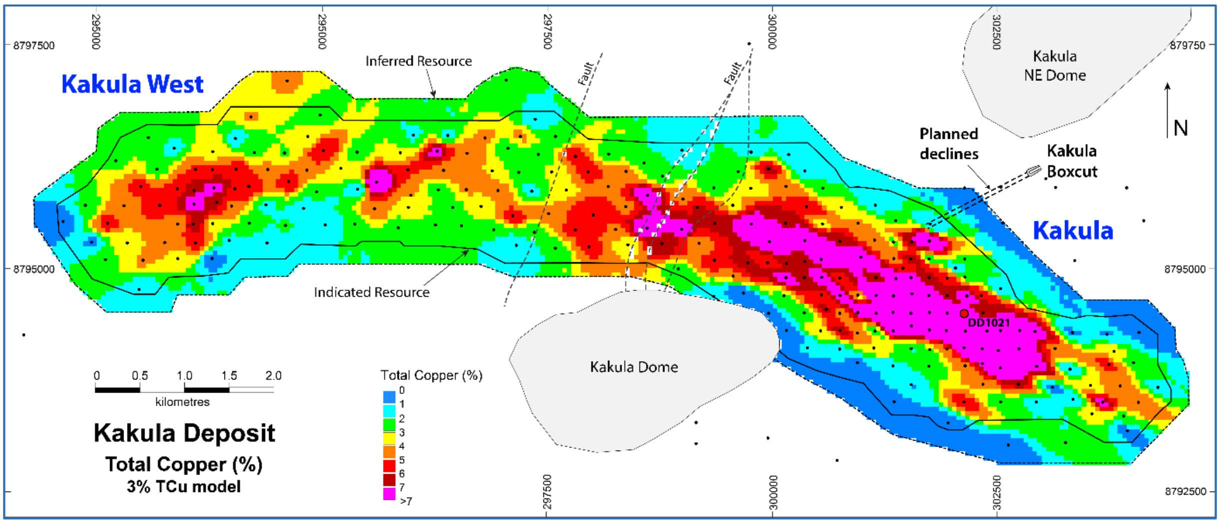


Figure provided by Ivanhoe, 2018. Missing blocks are due to fault offsets.

Figure 7.39 The Impact of Lithology on the Characteristics of the Grade Profile

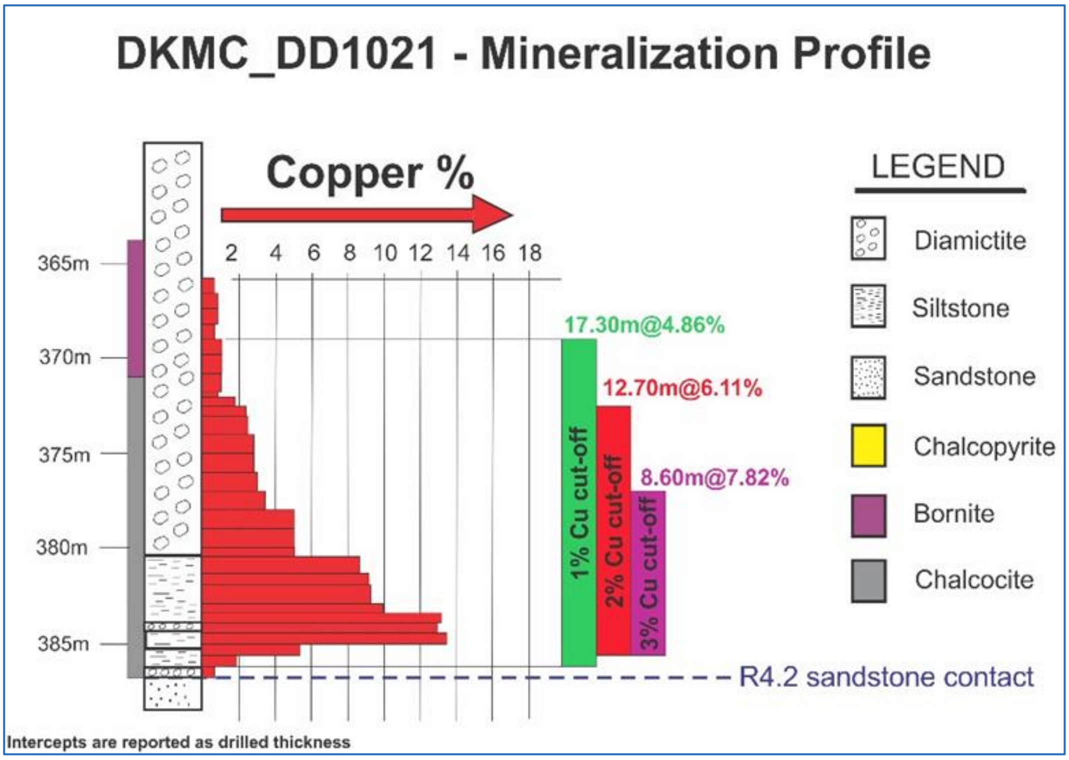


Figure provided by Ivanhoe, 2016.

In the central to south-east high-grade portions of Kakula, a strong relationship exists between the highest-grade intersections and the presence of the basal siltstone. Mineralised zones are thickest where the siltstone is thickest, and even within the gradually-weakening vertical grade profile, a sharp drop in grade can be observed at the top contact of the siltstone. A TCu (%) histogram of 1 m composite samples within the 3% grade shell shows a clear bimodal distribution. If the samples are separated based on a simplified host lithology (siltstone or diamictite), it becomes obvious that samples hosted in the siltstone account for the high-grade population, whilst the diamictite-hosted samples account for the lower-grade population (Figure 7.40).

Figure 7.40 The Bimodal TCu (%) Distribution is Easily Explained by the Distinction Between Host Lithologies at Kakula

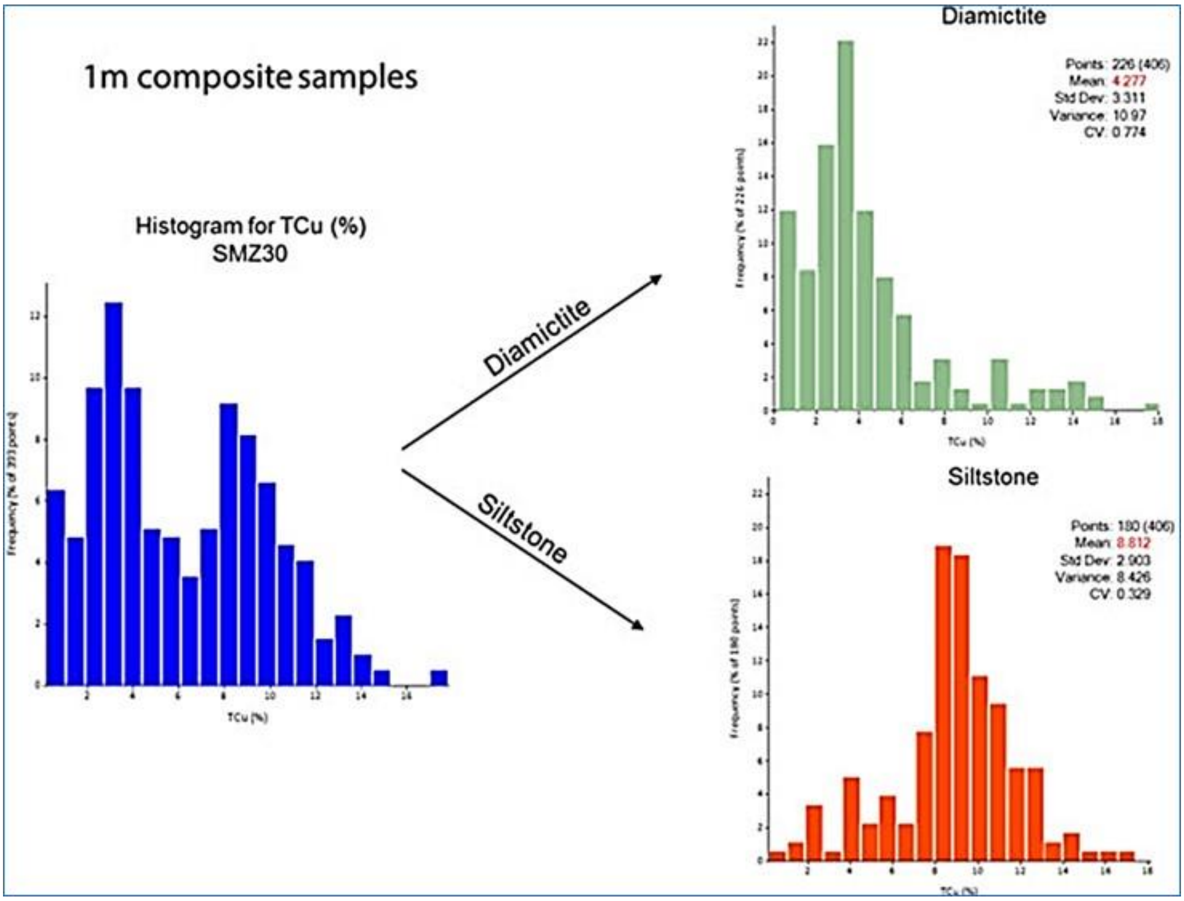


Figure provided by Ivanhoe, 2016.

A distinct maroon colour to the matrix of the diamictite has been observed in high-grade intersections. Investigations are still ongoing to explain this colour change, but it appears to highlight a control on the chalcocite mineralisation (Figure 7.41).

In the south-eastern portions of Kakula, the highest-grade intersections align very strongly along the 115° trend evident in the different stratigraphic and lithological units (refer to discussion in Section 7.3.1 and 7.3.2). To the north-west, the mineralisation turns to the west, with alignment along 105°. At Kakula West, well developed growth faults control the alignment of thickness and grade trends along 065°. The intensity of these controls and their incorporation into the grade estimation are discussed in Section 14.

Figure 7.41 A Typical Basal, High-grade Portion of a Kakula Intersection, Highlighting the Maroon Colour and Basal Siltstone

Figure provided by Ivanhoe, 2016.

Mosaic Patterns to Mineralisation

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

Figure 7.42 shows the November 2017 Kamoa Mineral Resource model for TCu and true thickness with superimposed drillholes. There are clear discontinuities in grade and thickness around mosaic pieces running greater than 3.5% TCu or having true thicknesses over 10 m.

Figure 7.42 TCu Grade (left) and Vertical Thickness (right) for the Kamoa Deposit 2017 Mineral Resource

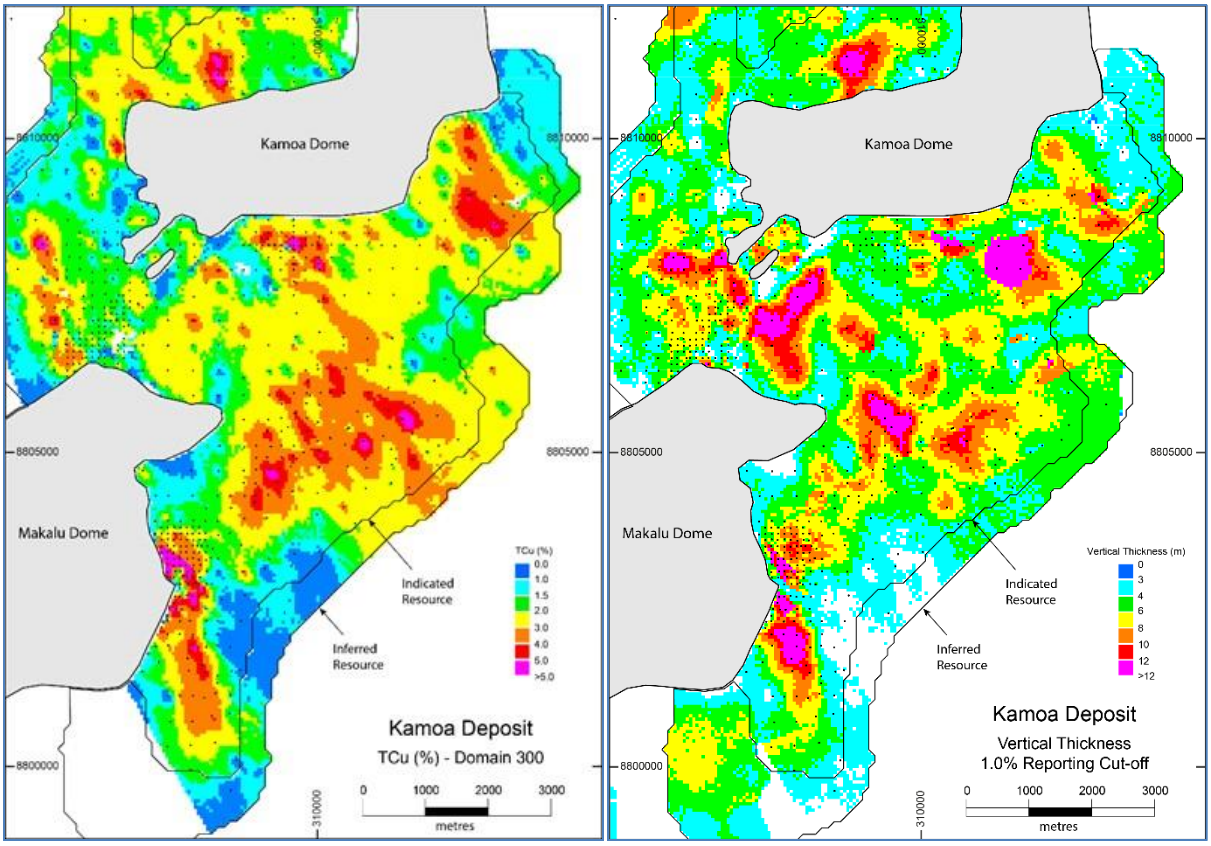
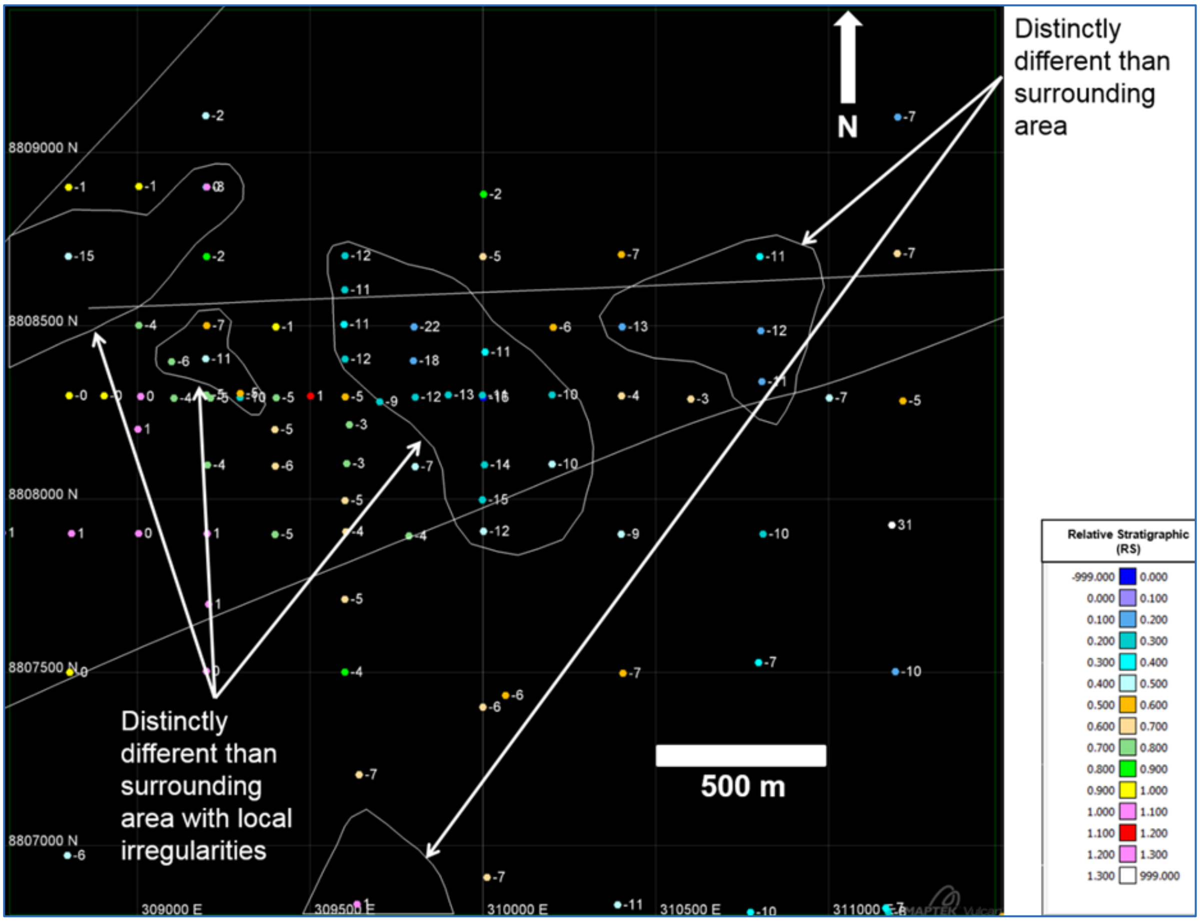


Figure provided by Ivanhoe, 2018. Average grade for blocks from the 3D model selected within the 2D SMZ10 wireframe for Domain 300 (Upper SMZ). Vertical thickness from the 3D model selected above a 1% reporting cut-off.

Figure 7.43 shows an area at the Kamoa deposit delineated using 200 m spaced drillholes. Plotted is the distance between the base of the KPS and the centroid of the SMZ10, the mineralised intercept at a 1%, TCu cut-off over 3 m used to select the mineralised zone (SMZ10). The stratigraphic position of the SMZ in relation to the bottom of the KPS unit and top of the Roan unit was reviewed by calculating the relative stratigraphic (RS) position {(RS=1- [(KPSz – SMZz)/KPSz – ROANz)]}. Again, discontinuities are present at the edges of mosaic pieces.

Figure 7.43 Stratigraphic Position of SMZ10 with Respect to the Base of the KPS



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

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

7.4 Comments on Section 7

The Amec Foster Wheeler geology and resources estimation QPs note the following:

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

8 DEPOSIT TYPES

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

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

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

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

8.1 Comments on Section 8

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

Key features of the deposits include:

- Laterally continuous, have been drill tested over an area of 27 km x 21 km.
- Associated with a 35 km-long regional structural corridor bounded by the West Scarp Fault and Kansoko Trend.
- Strong host-rock control and restriction of the mineralisation to a redox boundary zone between oxidised footwall haematitic sandstone and reduced, sulphidic host diamictites and siltstone-sandstone rocks.
- Presence of the replacement, blebby, and matrix textures that are typical of sediment-hosted copper deposits.
- Vertical zoning of disseminated copper sulphide minerals from chalcocite to bornite to chalcopyrite.
- Hypogene minerals are chalcopyrite, bornite and chalcocite, with the predominant copper sulphide species varying spatially throughout the deposit. For example, deep drilling along the Kansoko Trend has intersected mixtures of bornite and chalcocite. Mineralisation at Kakula is predominately chalcocite.
- Occurrence of very fine-grained, bedded, disseminated copper sulphides in the intermediate sandy siltstone unit (Ki1.1.1.2) within the basal diamictite, or within the basal siltstone at Kakula, is typical of Zambian Ore Shale- style mineralisation.

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

9 EXPLORATION

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

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

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

9.1 Grids and Surveys

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

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

9.2 Geological Mapping

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

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

9.3 Geochemical Sampling

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

9.4 Geophysics

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

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

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

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

Ground gravity data were collected from eight lines at Kakula in 2016 to aid in mapping of the Ki1.1.1-R4.2 contact. These data were calibrated against 7,327 core density measurements and six wireline logs and combined with the geology model for Kakula. An airborne gravity survey was conducted over the entire Project in January 2018. Results are still being processed.

In 2016 and 2017, Quick Log Geophysics conducted downhole surveys on 12 holes at Kakula. The data collected included logged full wave sonic, dual density, resistivity and gamma. Acoustic Televiewer (ATV) data were also obtained.

9.5 Petrology, Mineralogy, and Research Studies

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

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

The main conclusions from the study are:

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

Ivanhoe, through the Laurentian-Ivanhoe Mines Education partnership is part-funding two PhD research projects and three MSc research projects on Kamoa-Kakula. Areas or research include: Mineralising fluids of the Kamoa-Kakula deposits; The Geologic History of the diamictite matrix at Kamoa-Kakula; U-Pb geochronology of the Kamoa-Kakula host succession; Stratigraphic and geochemical controls on Kamoa-Kakula; Re-Os geochronology of the Kamoa-Kakula ore minerals.

9.6 Exploration Potential

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

Initial exploration programs identified a number of priority grass-roots exploration prospects within the Project, based on geological interpretations, stream-sediment and soil sampling, and aircore, RC, and core drilling. The most prospective area, Kakula, has been drill tested, modelled, and Mineral Resources have been estimated. A westerly extension to Kakula (Kaklua West) identified in 2017 has now also been drill tested, and is included in this Mineral Resource. A target for further exploration (Kamoa-Makalu) is discussed in Section 14.18.

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

9.7 Comments on Section 9

In the opinion of the Amec Foster Wheeler QPs:

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

10 DRILLING

10.1 Introduction

The drillhole database used for the Kamoa resource estimation was closed on 23 November 2015. The drillhole database used for the Kakula resource estimation was closed on 26 January 2018. Aircore, reverse circulation (RC) and core drilling have been undertaken since May 2006. Aircore and RC drilling were used in early exploration to follow up identified anomalies. None of these holes are used for resource estimation. Coreholes have been used for geological modelling, and those occurring within the mining lease and in areas of mineralisation (drillholes on the Kamoa and Makalu domes are excluded) have been used for resource estimation.

As at 21 February 2018, there were 1,587 coreholes drilled within the broader Project area (Table 10.1). The statistics in Table 10.1 are based on the current drillhole collar data provided to Amec Foster Wheeler in the form of an Excel spreadsheet by Ivanhoe (Gilchrist, 2018). Amec Foster Wheeler recognizes that the acQuire project data base is currently in process of being updated. The 2017 Kamoa Mineral Resource estimate used 776 drillhole intercepts. Included in the 776 drillholes are 16 twin holes (where the spacing between drillholes is <25 m) and six wedge holes. Although a far greater number of holes have been wedged, the wedges have typically been used in their entirety for metallurgical testing, and have thus not been sampled for resource estimation purposes. In these cases, only the parent hole is used during Mineral Resource estimation.

The 2018 Kakula Mineral Resource estimate used 271 drillhole intercepts. Only 254 of these occur within the Kakula Indicated and Inferred Resource model perimeter. Three drillholes are located south-west of Kakula West, five drillholes are located around the southern edge of the Inferred outline, and nine drillholes are located north-east of the Inferred outline.

The 540 holes not included in either the Kamoa or Kakula estimates were excluded because they were either abandoned, unmineralised holes in the dome areas, unsampled metallurgical, civil geotechnical or hydrological drillholes, or were drilled after the closure of the database for the Kamoa Mineral Resource estimation (23 November 2015) or after the closure of the database for the Kakula Mineral Resource estimation (26 January 2018), refer to Table 10.1.

Figure 10.1 shows the collar locations of drillholes occurring inside the Project area as at 21 February 2018. Figure 10.2 shows the completed drilling at Kakula as of 16 May 2017. Coreholes typically commence collecting cores at PQ size (85 mm), reducing to HQ size (63.5 mm), and where required by ground conditions, further reducing to NQ size (47.6 mm).

Table 10.1 Drilling Statistics per Drill Purpose for Coreholes (as at 21 February 2018)

Drill Purpose	Count (Active)	Metres (m)
	Resource	
Kamoa estimate (2017)	776	225,620.4
Kamoa (post-estimation)	30	8,991.9
Kakula estimate (2018)	271	148,929.7
Kakula (post-estimation)	19	10,095.8
Exploration	10	4,466.0
Domes	113	10,660.2
Metallurgy	116	13,645.3
Geotechnical	22	3,877.2
Civil Geotechnical	64	2,125.1
Condemnation	51	1,177.8
Cover Drilling	10	1,763.6
Permeability	5	30.0
Abandoned	100	19,958.3
Total	**1,587**	**451,341.3**

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

Figure 10.1 Mineral Resource Definition Drilling at Kamoa-Kakula

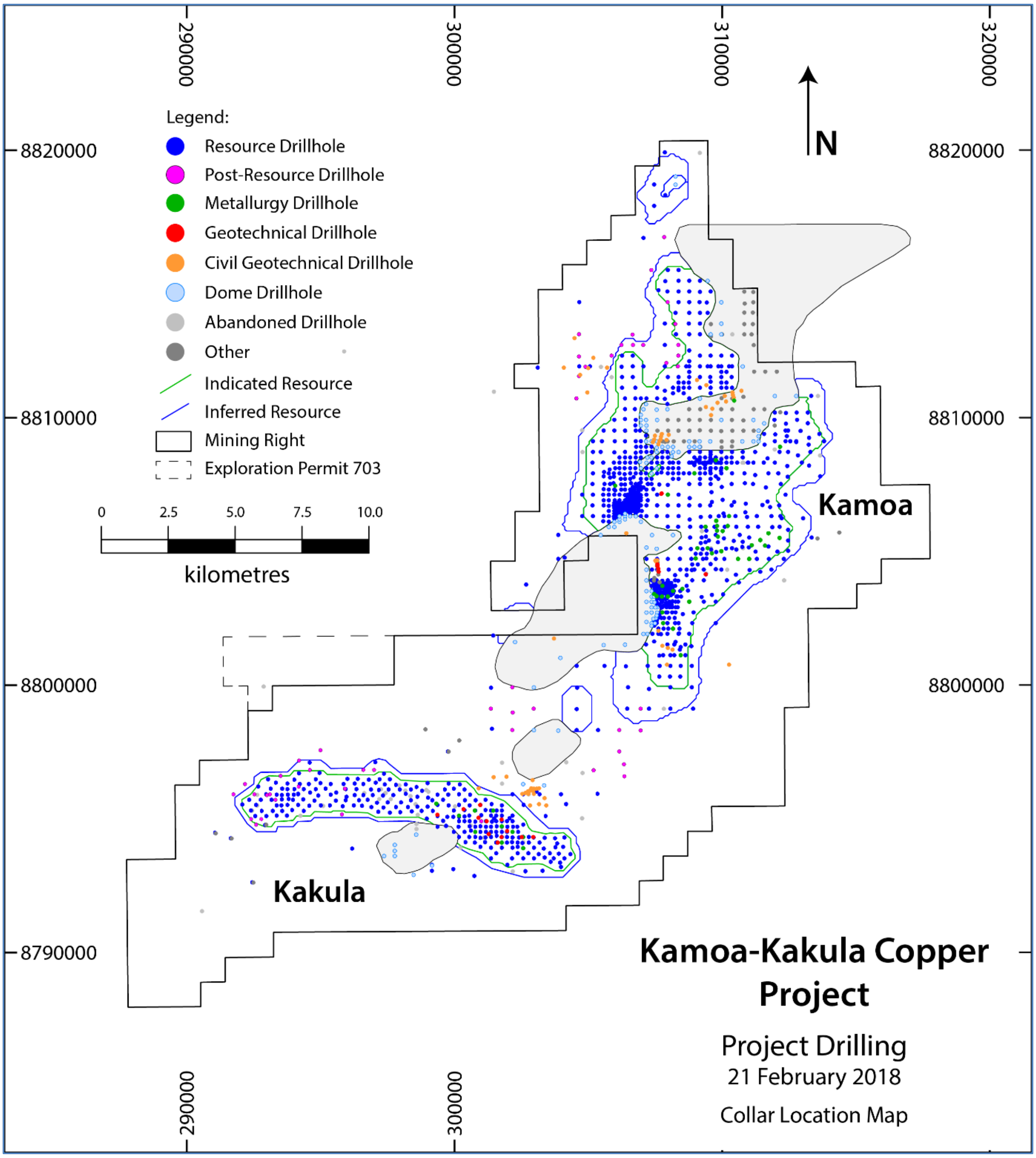


Figure provided by Ivanhoe, 2018. 'Other' includes exploration drillholes, condemnation drillholes, cover drillholes and permeability drillholes.

Figure 10.2 Drill Location Plan, Kakula

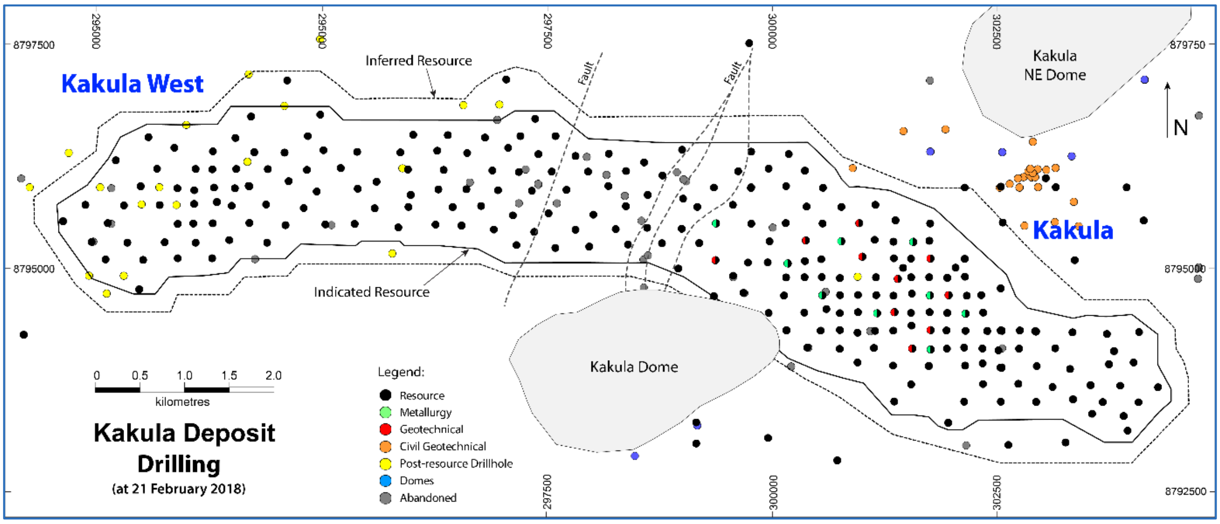


Figure provided by Ivanhoe, 2018. Positions shown are end-of-hole co-ordinate positions.

10.2 Geological Logging

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

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

RC drill chips were logged at the drill site, and representative samples are stored in chip trays for each 1 m interval. Samples at the base of the aircore holes were also retained for reference. Coreholes were logged at the core shed located in Kolwezi until 2009; following this all logging was moved to the Kamoa drill camp.

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

During 2012, a new logging scheme was implemented to streamline the logging process to begin collecting data more relevant to future mining activities, and to record logged data on mini-laptops. This eliminates delays and errors associated with data entry of paper logging forms.

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

10.3 Core Handling

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

10.4 Recovery

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

Core recovery data at Kakula are generally very good. However, a review of the database indicates there are a number of intervals with recoveries well over 100%; these intervals are currently being reviewed by Ivanhoe staff.

10.5 Collar Surveys

All drill sites are initially surveyed using a hand-held GPS that is typically accurate to within about 7 m. Prior to finalisation of a resource database, all outstanding collar surveys for completed holes that are to be included in the estimate are surveyed by an independent professional surveyor, SD Geomatique or E.M.K. Construction SARL, using a differential GPS which is accurate to within 20 mm. As of 26 January 2018, there were five drillholes remaining to be surveyed at Kakula, with two of these (DKMC_DD1228 and DKMC_DD1299) being used in the Mineral Resource estimate. All collars for holes used in the Kamoa Mineral Resource estimate were surveyed.

10.6 Downhole Surveys

10.6.1 Kamoa

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

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

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

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

10.6.2 Kakula

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

A total of 1,242 drillholes of the 271 holes used in the Kamoa resource modelling have an initial inclination of -80 to -90°, ranging in total depth from 54 m to 1,448.5 m. The remaining 29 drillholes had initial inclinations ranging from -79° to -63°, and these holes have total depths ranging from 55 m to 952 m.

10.7 Geotechnical Drilling

Where core is sufficiently competent to allow orientation surveys to be performed, Ivanhoe collects structural information for geotechnical and geological studies. The location and purpose of geotechnical drillholes at Kamoa are detailed in the January 2017 Technical Report.

As of 21 February 2018, the database contained 22 drillholes drilled exclusively for geotechnical purposes (3,877.2 m) and 64 civil geotechnical drillholes (2,125.1 m). These geotechnical holes can form part of the resource infill grid, or are drilled as separate wedges. When drilled as part of the resource infill grid, each one had a HQ wedge of the mineralised zone that has been sampled and assayed and included in the resource estimation; the parent intersection was specifically used for rock mass characterisation testwork. The holes were drilled at various azimuths at dips between -44° and -90°.

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

10.8 Hydrogeological Drilling

The location of hydrogeological drillholes at Kamoa are detailed in the Kamoa 2016 PEA. No hydrogeological drillholes have been completed at Kakula.

10.9 Metallurgical Drilling

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

10.10 Sample Length/ True Thickness

At Kakula, a comparison of estimated true thickness to the drillhole composite length within the 3% TCu grade shell shows the estimated true thickness to be about 4% less than the drillhole intercept (Figure 10.3). The largest difference is apparent in DKMC_DD1051W1, which has an inclination of approximately -56°.

Figure 10.3 Kakula Drillhole Intercept Length Versus Estimated True Thickness

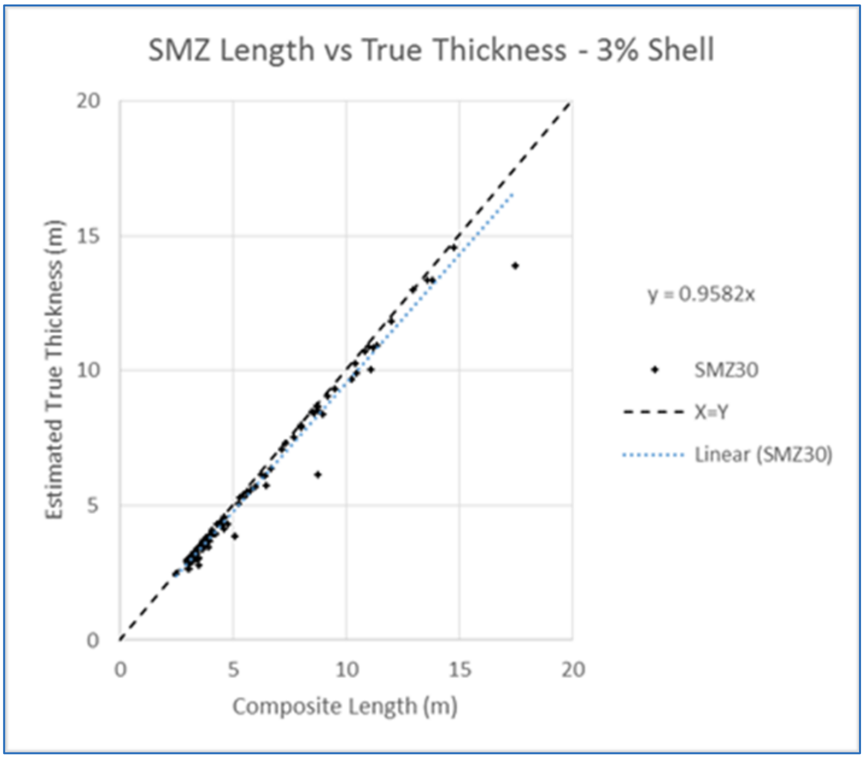


Figure prepared by Amec Foster Wheeler, 2017.

10.11 Drilling Since the Mineral Resource Database Close-off Date

10.11.1 Kamoa

The database contains 30 drillholes that post-date the Kamoa resource estimate database close-off date of 23 November 2015. Four of these holes were drilled in 2015, and 26 were drilled in 2017. Assays are now available for 23 of these drillholes, and these were not used in the 2017 Mineral Resource estimate. All of these holes were drilled for resource purposes, either as infill drillholes, or resource expansion drillholes.

Of the 30 drillholes completed after the Kamoa database close-off date, two were drilled at Kansoko Sud, two were drilled at Kansoko Nord, eight were drilled south of the Makalu Dome, and 18 were drilled in the Kamoa Ouest/Kamoa Nord area.

Although the newer drilling within the resource modelling area will change the grades locally, overall the new drilling should have a minimal effect on the average grade of the model. Figure 10.4 shows the results of holes drilled at Kamoa Ouest/Kamoa Nord (16 of the 23 holes with assay results). Figure 10.5 shows the location of the drillholes completed since the resource model.

Table 10.2 shows assay results and thicknesses for seven SMZ intercepts from the new drilling. The composite intervals shown usually do not include internal intervals of lower-grade material as is commonly found in other deposit types. The change in grade from non-mineralised to >1% Cu is usually distinct, and within the mineralised zone, grades typically remain above the 1% Cu over the entire intercept. This consistency of grade is typical of the Zambian Copperbelt deposits. If the SMZ could not be composited to meet the 1% Cu cut-off over a 3 m length criterion, a 3 m length with the highest copper grade available in the appropriate stratigraphic position was used to form the SMZ.

The QPs consider that the new drilling at the Kamoa deposit should have limited effects on the average grade of the deposit within the area of the currently-estimated Mineral Resources.

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

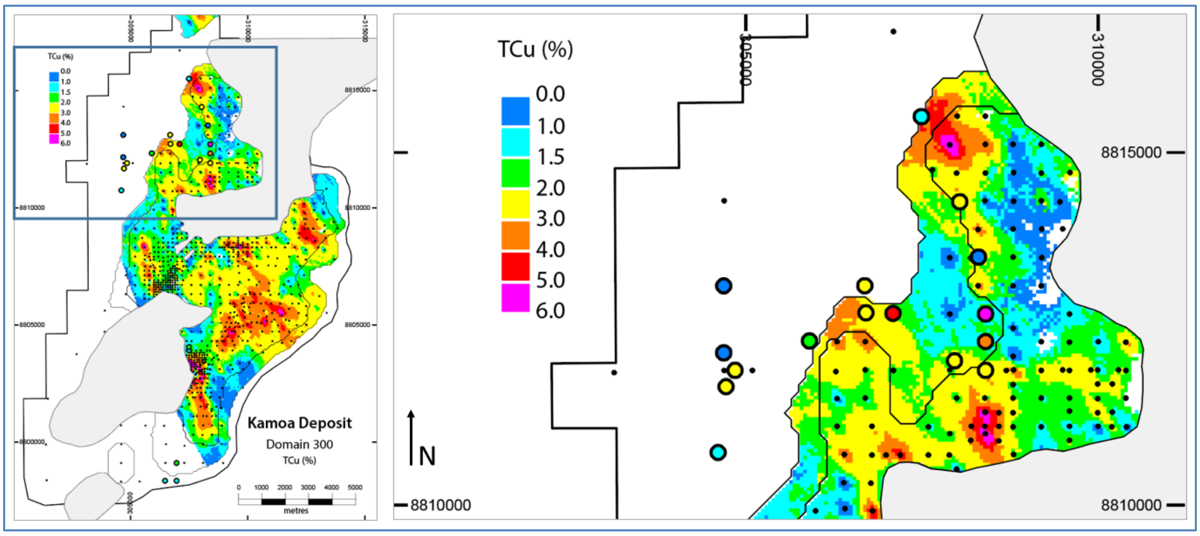


Figure by Ivanhoe, 2018. 2017 Resource model is in the background; both blocks and post-resource drillholes have been color-coded according to the legend; smaller black circles represent previously drilled holes used in the resource model.

Figure 10.5 Plan View Showing Kamoa Drillholes Completed Since Construction of the 2017 Mineral Resource Model

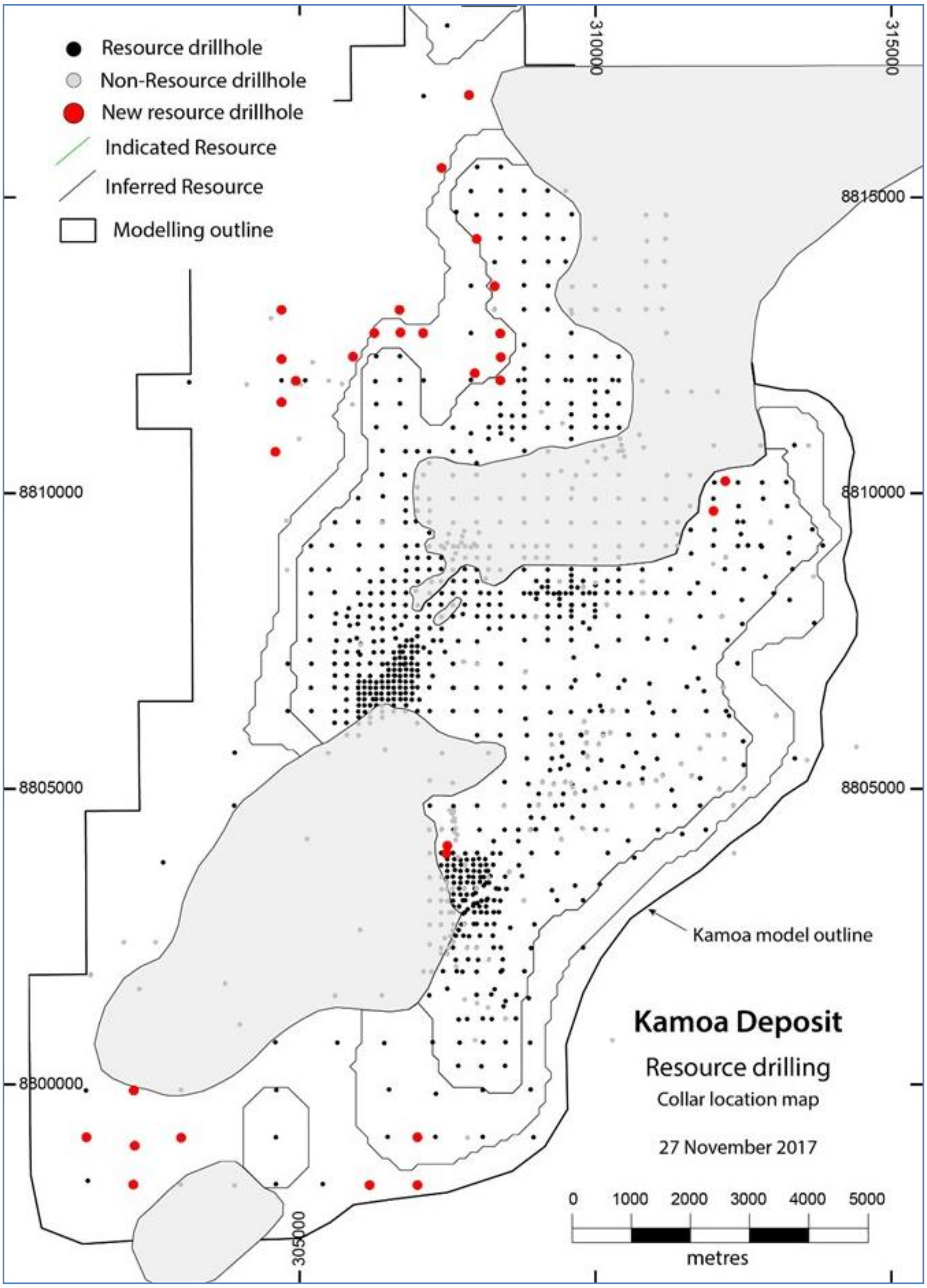


Figure provided by Ivanhoe, 2017.

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

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

10.11.2 Kakula

Between 26 January 2018 and 21 February 2018, Ivanplats completed an additional 10 core drillholes (6,107.9 m) in the Kakula area. The collar locations of the coreholes are shown in Figure 10.6. The core drillholes were drilled for exploration and infill purposes. Assays have yet to be received for any of these new drillholes.

Figure 10.6 Core Drilling Completed at Kakula after 26 January 2018 (as at 21 February 2018)

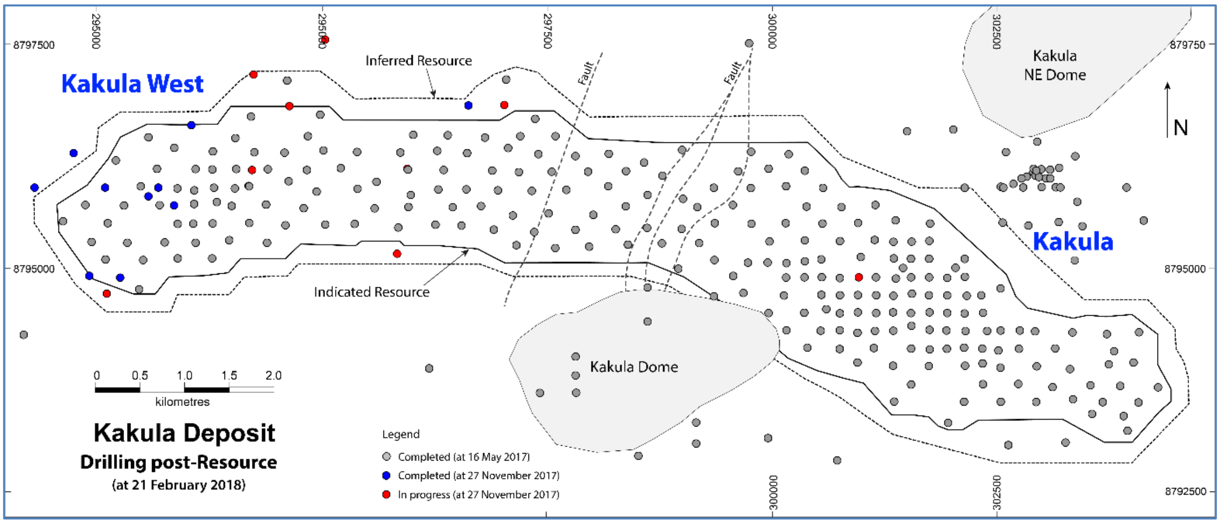


Figure provided by Ivanhoe, 2018. Note the cluster of drillholes to the north-east of the deposit are geotechnical holes for the portal of the box-cut.

The impact of the new drilling on the currently estimated Mineral Resource at the Kakula deposit cannot be ascertained by the QPs at this time as no assay results are available for these new drillholes.

10.12 Comments on Section 10

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

- The drillhole collar purpose information in the acQuire database is currently being updated and needs to be reviewed in the next drillhole audit.
- Drill intersections, due to the orientation of the drillholes, are typically slightly greater than the true thickness of the mineralisation.
- Drillhole orientations are generally appropriate for the mineralisation style.
- Core logging meets industry standards for sediment-hosted copper exploration.
- Collar surveys were performed using industry-standard instrumentation.
- Downhole surveys provide appropriate representation of the trajectories of the coreholes.
- Core recoveries are typically excellent. Ivanhoe is reviewing some instances of core recoveries in excess of 100% at Kakula.
- The SMZ can include both lower- and higher-grade mineralisation; however, the transition in grade from non-mineralised to >1% Cu is usually distinct, and within the mineralised zone, grades typically remain above 1% Cu over the entire intercept.
- No material factors were identified with the data collection from the drill programmes that could affect Mineral Resource estimation.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Witness Sampling

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

11.2 Sampling Methods

11.3 Geochemical Sampling

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

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

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

11.3.1 RC Sampling

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

11.3.2 Core Sampling

The core sampling procedure is as follows:

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

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

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

11.4 Metallurgical Sampling

11.4.1 Kamoa

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

The Xstrata Process Support (XPS) metallurgical samples were half HQ core; the core was then individually crushed to -3.36 mm topsize, followed by blending and sub-sampling by spinning riffler into 2.0 kg replicate test charges.

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

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

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

11.4.2 Kakula

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

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

11.5 Specific Gravity Determinations

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

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

At Kakula, a total of 13,540 SG measurements were performed on samples obtained from remaining half core after the other half was prepared and sent to Bureau Veritas for analysis. Of these measurements, there are 13,537 samples with SG values between 1.5 and 5.0.

11.6 Analytical and Test Laboratories

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

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

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

ALS of Vancouver, British Columbia, acted as the independent check laboratory for drill core samples from part of the 2009 program and for 2010 through 2017 drilling. ALS is ISO: 9001:2008 registered and ISO:17025-accredited.

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

Table 11.1 Analytical Laboratories Used

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

11.7 Sample Preparation and Analysis

A mobile sample preparation facility housed in a shipping container was purchased in Zambia in 2006, and relocated to Kolwezi in the DRC. This facility contains two jaw crushers and two LM2 puck-and-bowl pulverisers. The laboratory is managed by Ivanhoe personnel and was monitored by Richard Carver of GCXplore Ltd. between 2006 and 2009. All drill core samples collected prior to November 2010 were processed by the Kolwezi facility; subsequently (since drillhole DKMC_DD209) they have been processed at the Kamoa site facility. The equipment at the facility includes two TM Terminator Jaw crushers, two Labtech Essa LM-2 pulverisers and two riffle splitters.

Sawn drill core is sampled on 1 m intervals, and then the sawn core is crushed to nominal 2 mm using jaw crushers. A quarter split (500 g to 1,000 g) is pulverised to >90% -75 µm, using the LM2 puck and bowl pulverisers. A 100 g split is sent for assay; three 50 g samples are kept as government witness samples, 30 g for Niton analysis, and approximately 80 g of pulp is retained as a reference sample. The remaining coarse reject material is retained.

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

11.8 Sample Analysis

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

11.8.1 Bureau Veritas (formerly Ultra Trace) Laboratory

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

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

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

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

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

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

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

11.9 Quality Assurance and Quality Control

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

11.9.1 Blanks

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

11.9.1.1 Kamoa

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

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

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

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

11.9.1.2 Kakula

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

11.9.2 Duplicates

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

11.9.3 Certified Reference Materials

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

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

11.10 Databases

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

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

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

11.11 Sample Security

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

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

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

11.12 Comments on Section 11

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

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

12 DATA VERIFICATION

12.1 Amec Foster Wheeler Verifications (2009–2018)

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

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

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

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

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

12.2 QA/QC Review

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

12.2.1 Kamoa Screen Tests (2009–2013)

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

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

12.2.2 Kamoa Certified Reference Materials (2009–2013)

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

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

12.2.3 Kamoa Check Assays (2009–2014)

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

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

12.2.4 Kamoa Duplicate Assays (2009–2013)

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

12.2.5 Blanks (2009–2013)

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

12.2.6 2014 Kamoa QA/QC Review

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

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

Coarse reject duplicate results indicate adequate precision.

Blank samples do not indicate any sample contamination.

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

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

12.2.7 Kamoa Acid Soluble Copper Determinations

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

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

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

12.2.8 Kakula QA/QC Review

Amec Foster Wheeler conducted a high-level review of the initial memorandum provided by Acuity Geoscience Ltd. (Acuity, 2016) discussing Acuity's review of the Kakula QA/QC data available as of 9 October 2016. Acuity prepared a second memorandum (Acuity, 2017) covering the period 1 July 2014 to 20 May 2017. All charts and graphs shown in this section were generated by Acuity. In February 2018, Acuity prepared a memorandum (Acuity, 2018a) covering the period 21 May 2017 to 28 January 2018. All charts and graphs shown in this section were generated by Acuity.

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

Coarse reject duplicate results indicate adequate precision.

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

Kakula CRMs with grades greater than 5.31% Cu returned all values well within the ± 2SD tolerance limits. At and below 5.31%, several quality control monitoring failures are related to a slight positive bias. These failures affect only a small number of adjacent routine samples, whose grades are below those of the high-grade zones in the current mineral resource estimate. Therefore, the effect of the failures on the overall quality of data is negligible.

CRM results typically indicate biases much less than 5%. Table 12.1 summarises the Kakula CRM results; Figure 12.1 shows the performance charts for KAM01, KAM02, KAM03, KAM04, KAM05, and KAM06. The Kakula assay results (RT) are shown on the right portion (BVM) of the charts. The round robin (RR) results are shown on the left portion of the charts.

Table 12.1 Kakula CRM Results

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

CRM Range	CRM Name	RR Cu ppm	RR RSD %	CRM Count	BVM Cu ppm	BVM Bias %	BVM RSD %
Ext. >10%	KAM06	151979	2.80	11	150182	-1.183	1.22
	KAM05	100545	2.32	30	100260	-0.278	1.12
V. High 5-10%	OREAS97	63100	5.39	6	63000	-0.158	0.72
	KAM04	53100	1.79	50	53140	0.07	1.07
High 2-5%	KAM03	48159	1.91	52	48123	-0.075	0.83
	OREAS96	39300	2.80	14	39186	-0.291	1.28
	OREAS95	25900	2.51	24	25633	-1.03	1.10
	KAM02	23214	2.07	85	23108	-0.456	0.95
Med. 1-2%	KAM01	12750	2.07	106	12788	0.131	1.30
Barren <500 ppm	BLANK2010	~15	NA	15	12	NA	NA
	BLANK2014	~15	NA	247	10	NA	NA
	OREAS22D	9	NA	73	10	NA	NA

Figure 12.1 CRM Performance Chart for Kakula Matrix Matched Reference Materials

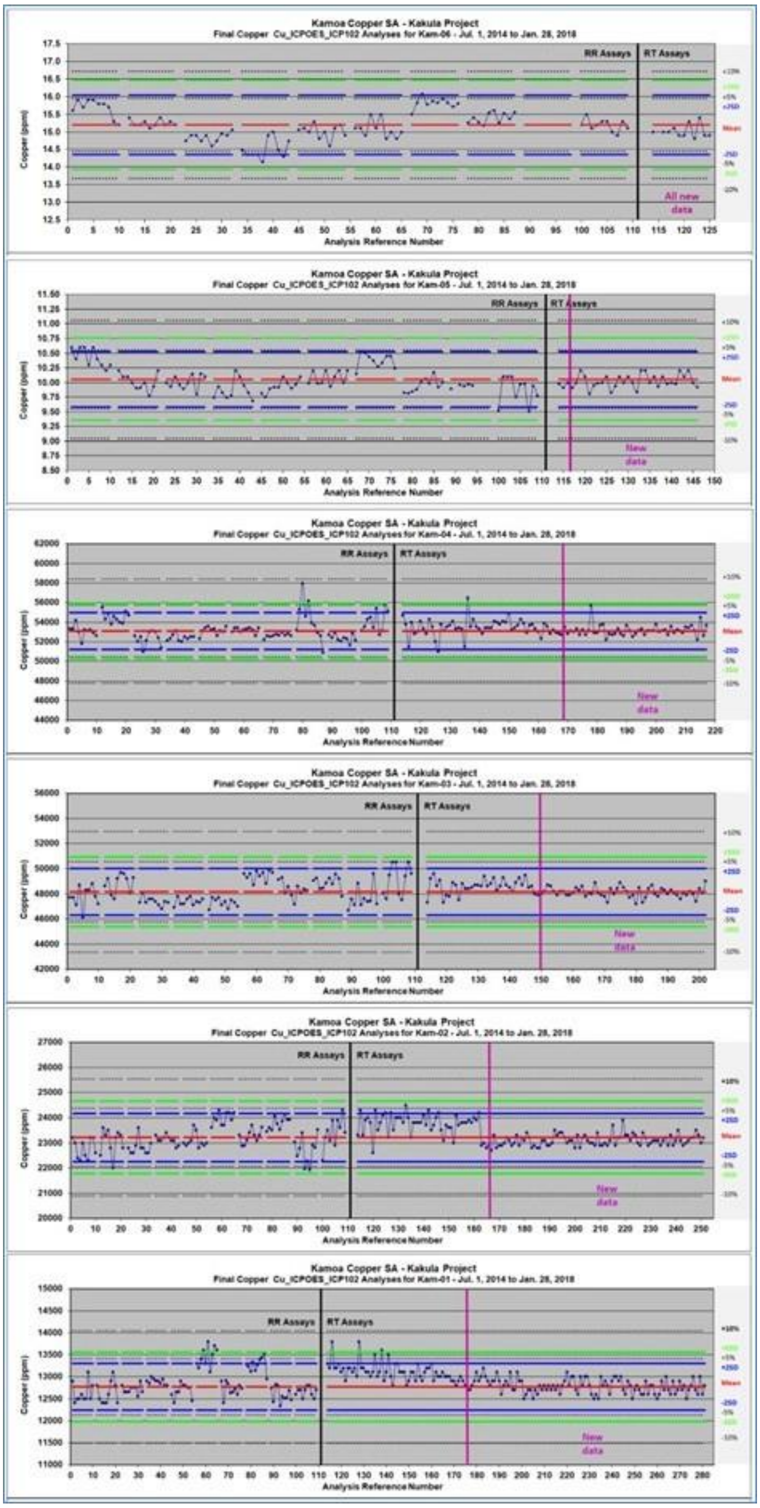


Figure by Acuity, 2018.

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

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

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

12.3 Site Visits

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

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

12.3.1 Field Drill Collar Check

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

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

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

12.3.2 Drilling and Core Storage

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

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

In 2010, a new core-logging facility and new secure core-storage facility were constructed at the Kamoa site. As of July 2010, all new core samples are stored at the facility. Figure 12.2 and Figure 12.3 show the logging facility and core storage respectively.

Figure 12.2 On-Site Core-Logging Facility

Photograph by Amec Foster Wheeler, 2011.

Figure 12.3 On-Site Core-Storage Facility

Photograph by Amec Foster Wheeler, 2011.

12.3.3 Inspection of Drill Core

The following Kamoa coreholes were examined during the 2009 visit:

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

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

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

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

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

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

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

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

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

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

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

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

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

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

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

12.3.4 Sample Preparation Facilities

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

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

12.4 Copper Grade Check Sampling

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

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

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

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

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

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

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

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

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

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

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

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

12.5 Comments of Section 12

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

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

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Testwork Overview

Between 2010 and 2015 a series of metallurgical testwork programs were completed on drill core samples of known Kamoa copper mineralisation. These investigations focussed on metallurgical characterisation and flowsheet development for the processing of hypogene and supergene copper mineralisation. 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 mineralisation expanded progressively, and this led to major changes to mine schedules and associated processing schedules. As an example, over time the supergene mineralisation became less important and the testing focus shifted to hypogene mineralisation. Another example is that the resource and reserve grades increased as better mineralised zones were identified. Such learning and transitions are not uncommon for this style of mineralisation. The historic sample selection and testwork, defined as Phases 1 to 5, provided the requisite metallurgical understanding to support the 2012 PEA and subsequent Technical Reports ahead of the Kamoa 2017 PFS.

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

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

During 2016, Kamoa Copper SA discovered the Kakula deposit that has significantly higher copper head grades than the Kamoa deposit. It was decided that fast-tracking the Kakula Mine could have positive impact on the economics of the overall Kamoa-Kakula Project. The Kakula mine portal will be located about 11 km south-west of the Kansoko mine portal.

The initial metallurgical testwork on Kakula mineralisation was carried out in July 2016 using drill cores from early holes DD996 and DD998. The samples were sent to Zijin laboratories, in China for preliminary flotation tests. At Zijin, the samples were crushed and split in two equal halves, the other half was sent to the XPS laboratory in Canada to perform a confirmatory flotation tests. The scope of work for both laboratories included head analysis, grind calibration, and duplicate float tests on each of the core as well as on a composite of the two cores using the IFS4A PFS flowsheet developed at XPS.

The DD996 and DD998 samples (July 2016) showed a relative low head grade relative to the mine plan head grades, with a composite averaging only 4.1% copper head grade. The flotation test results at Zijin achieved a copper recovery of 86% and produced a concentrate with a grade of 53% copper. The flotation results indicate that satisfactory grades and copper recovery can be achieved using the IFS4A PFS flowsheet. The results also indicate that material from the Kakula and Kansoko zones could be processed through the same concentrator plant, which could yield significant operational and economic efficiencies.

Kakula mineralisation is consistently chalcocite rich, and not as variable when compared to other Kamoa material studied to date which contained varying proportions of bornite, chalcopyrite and chalcocite for the different areas.

Additional samples (September 2016) from drillholes DD1005 and DD1007 were sent to Zijin, and DD1012 and DD1036 samples were sent to XPS to verify metallurgical characteristics on a higher-grade sample to be analysed and tested to reconfirm if the Kakula mineralisation is compatible with the selected IFS4A Kamoa flowsheet.

This latest bench-scale metallurgical flotation testwork carried out at XPS laboratories on an 8.1% copper head grade sample, achieved copper recoveries of 87.8% and a concentrate grade with 56% copper at 12.5% mass pull. In addition, the arsenic in final concentrate was lower than achieved in the previous Kamoa test results. The material tested and the subsequent plant design will accommodate this 8.1% copper feed grade. Thus, both Zijin and XPS on different composite samples achieved similar recoveries and grades.

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

13.2 Historic Testwork Phase Definitions

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

Table 13.1 Kamoa Historical Metallurgical Testwork

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

13.3 Historical Metallurgical Sample Locations

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

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, they were used for flotation flowsheet 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 because separate hangingwall and footwall samples were sourced from one of the five holes.

Figure 13.1 Drill Collars for Metallurgical Test Phases 1 to 5

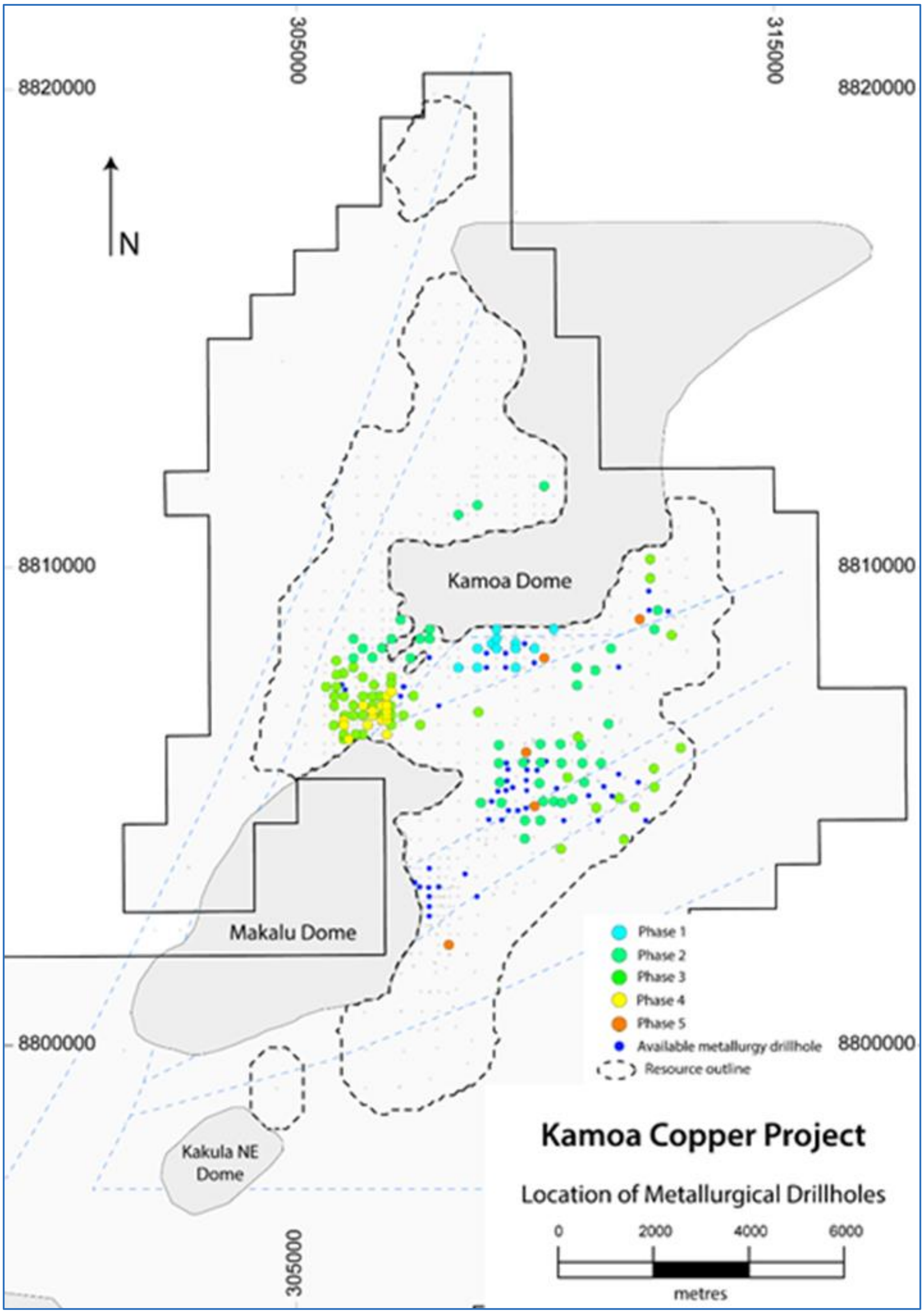


Figure provided by Ivanhoe, 2016.

13.4 Historical Comminution Testwork

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

Table 13.2 Historical Comminution Program, Sample Numbers Tested

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

13.4.1 Competence (SMC Test) Summary

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

Table 13.3 SMC Test Results as Axb Value Range

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

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

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

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

13.4.2 Fine Grindability (BBWI) Summary

The Bond Ball Mill Work Index test (BBWI) measures how difficult the sample is to grind from about 2 to 3 mm down to the 100 µm range. The index itself is a measure of the energy (in kWh/t) required to reduce the rock from infinite size to 100 µm P_{80}.

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

Table 13.4 BBWI Test Results as kWh/t Value Range

Phase	Diamictites (Hypogene, Supergene and unmineralised)				Oxide		Pyritic Siltstone (mineralised and unmineralised, hangingwall)			Sandstone (footwall)	
Closing Screen (µm)	212	106	75	53	106	53	106	75	53	106	53
1	–	15.5	15.7	–	–	–	16.3	14.6	–	–	–
2	–	13–17	–	–	–	–	17–20	–	–	16	–
4	–	–	–	–	11–13	11.5–14	–	–	–	–	–
5	20	14.5–22	–	13.5–21	–	–	15.1	–	13.4	14.5	15.2

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

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

13.4.3 Coarse Grindability (BBWI) Summary

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

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

Table 13.5 BRWI Test Results as kWh/t Value Range

Phase	Diamictites (Hypogene, Supergene and unmineralised)	Oxide	Pyritic Siltstone (mineralised and unmineralised, hangingwall)	Sandstone (unmineralised, footwall)
1	17–19	–	20.5	–
2	17–20	–	24	20
4	–	14	–	–
5	18–22	–	16.1	15.7

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

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

13.4.4 Crushability (CWI) Summary

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

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

Table 13.6 CWI Test Results as kWh/t Value Range

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

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

13.4.5 Abrasiveness (Ai) Summary

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

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

Table 13.7 Ai Test Results Value Range

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

The diamictites and the pyritic siltstone typically have Ai values less than 0.15 and all are below 0.25. These results indicate very low to low abrasiveness. The oxides also have low abrasion indices. The only sample with a high level of abrasiveness was sandstone. The siltstone and diamictites have very fine grain sizes and tend to act like polishing powder, while the sandstone has coarse quartz grains and acts in a manner similar to coarse sandpaper.

13.4.6 Historical Comminution Characterisation Summary

The four comminution properties measured are summarised in Table 13.8.

Table 13.8 Comminution Summary by Mineralisation Type

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

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

The favourable abrasiveness values in mineralised material mean the ball and liner consumptions will be low. Avoidance of the abrasive footwall sandstone during mining is strongly recommended.

13.5 Historical Flotation Testwork

13.5.1 Phase 1 (2010) – Mintek Laboratories South Africa

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

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

This Phase 1 flotation program indicated:

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

Figure 13.2 MF2 Dual Regrind Circuit Flowsheet

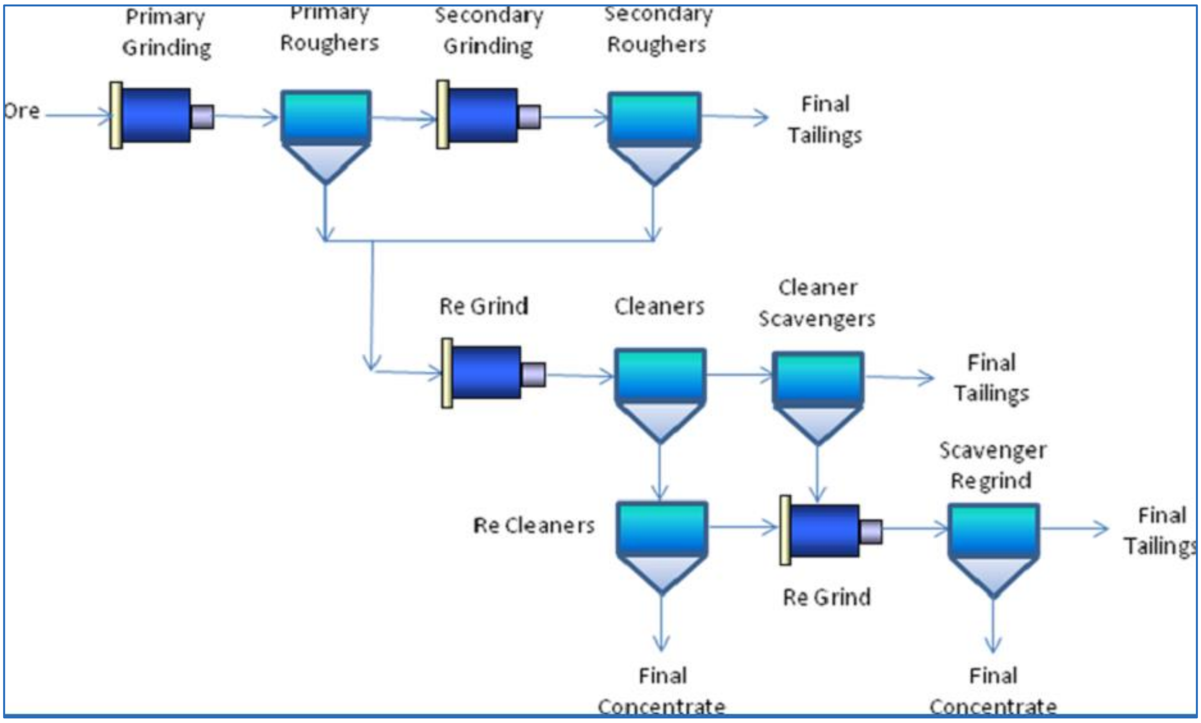


Image courtesy of Mintek 2010.

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

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

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

Phase 2 testing showed the:

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

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

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

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

A testwork program 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 program 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.

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

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

Figure 13.3 The Milestone Flowsheet

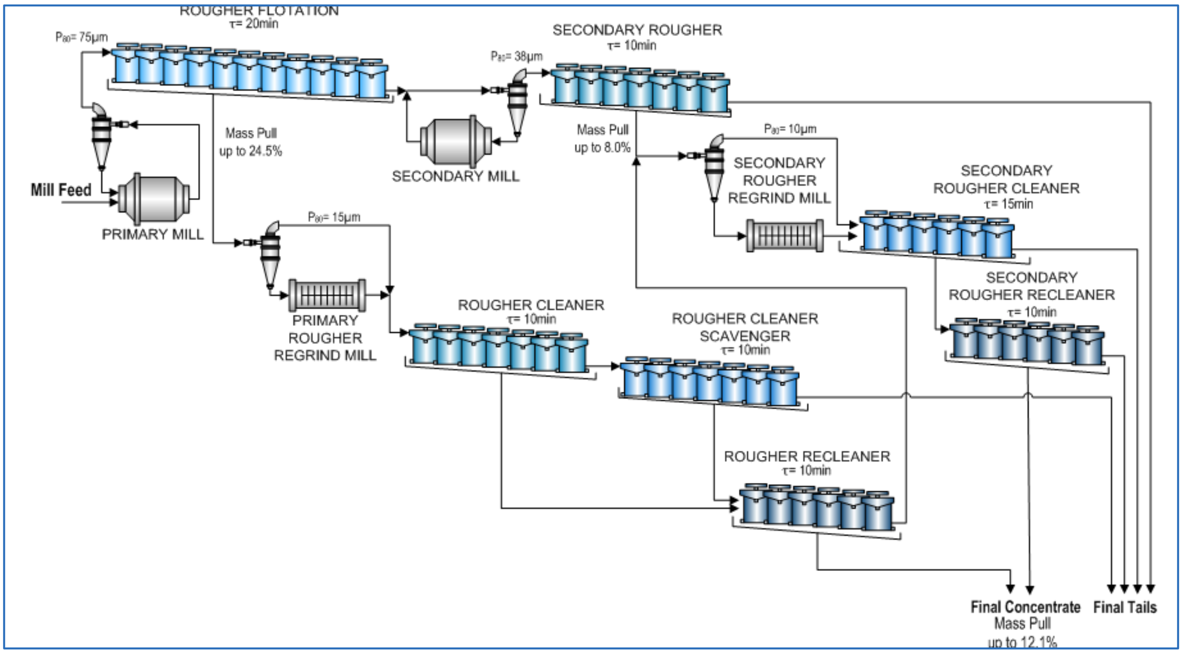


Figure by Hatch, 2013.

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 flowsheet from this work stage was termed the "Frozen Flowsheet" by XPS and is shown in Figure 13.4.

Figure 13.4 XPS Frozen Flowsheet

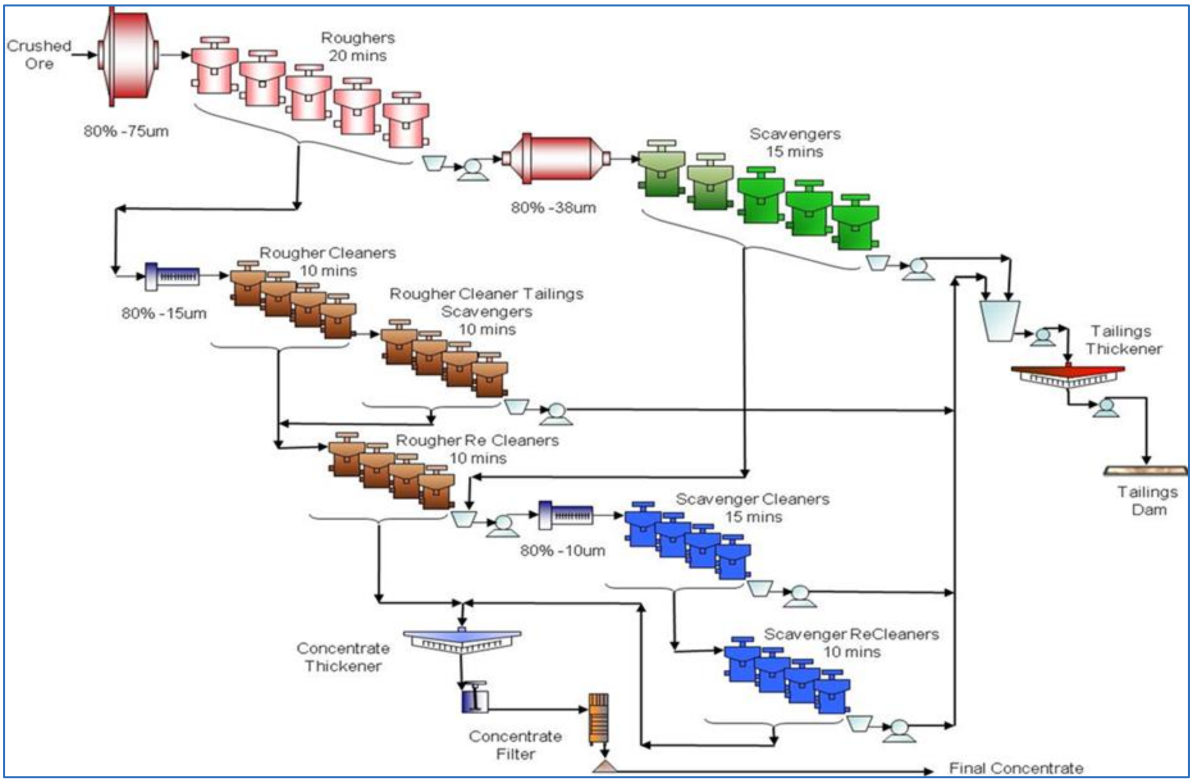


Image courtesy XPS, 2013.

This Phase 3 testwork program indicated:

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

13.5.4 Phase 4 XPS Flotation Testing

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

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

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

13.5.5 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 flowsheet. Mintek was used to verify the transferability of the XPS Frozen Flowsheet and to explore some additional process options.

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

Table 13.9 Comparison of Test Procedure at Two Laboratories

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

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

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

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

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

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

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

13.6 Kamoa 2017 PFS Design Testwork

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

13.6.1 Phase 6 Comminution Testwork – Mintek

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

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

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

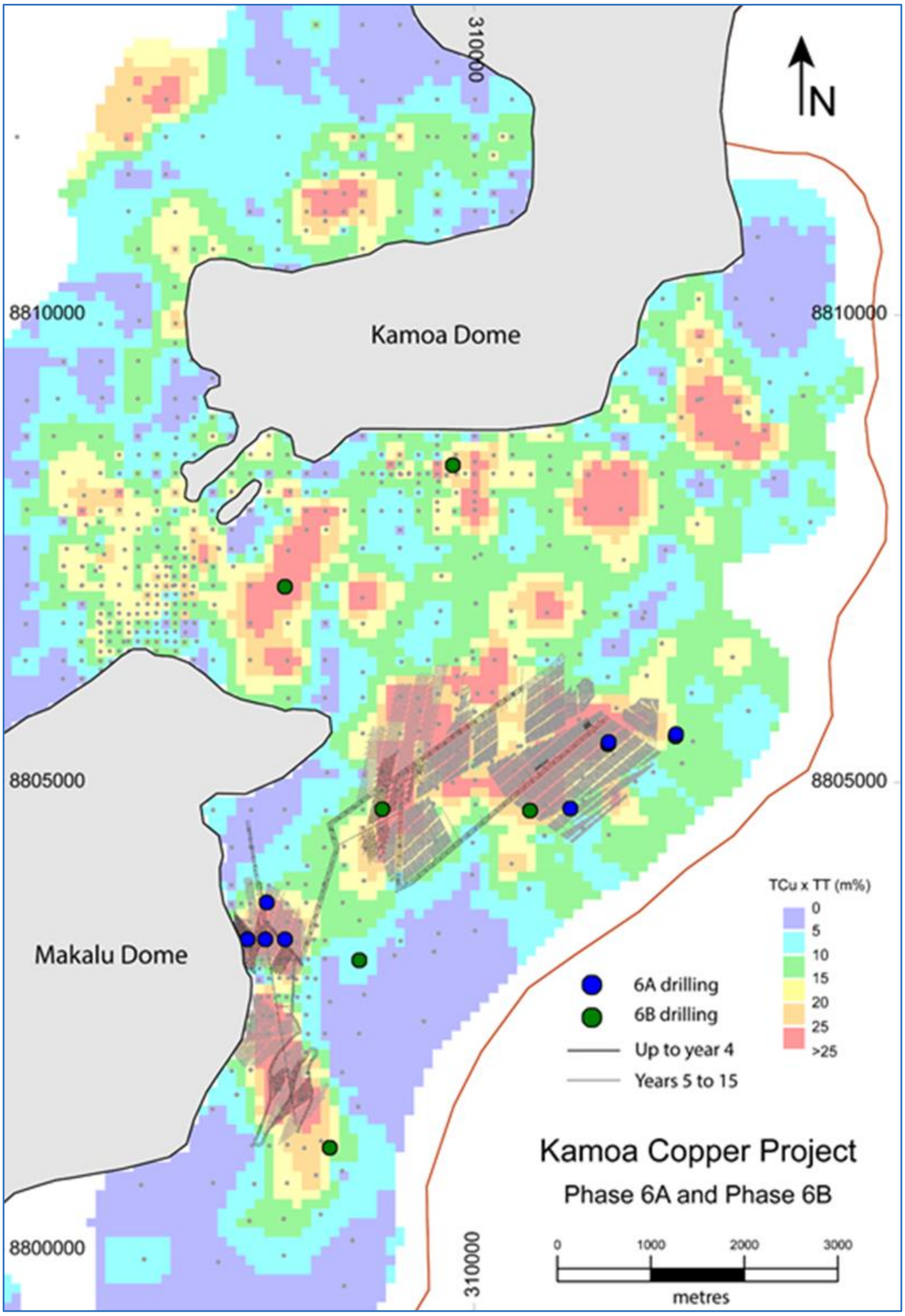


Figure provided by Ivanhoe, 2016.

Table 13.10 Phase 6 Comminution Summary

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

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

Table 13.11 Comminution Properties

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

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

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

Table 13.12 Design Comminution Properties

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

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

Figure 13.6 UCL90 Determination for Ai

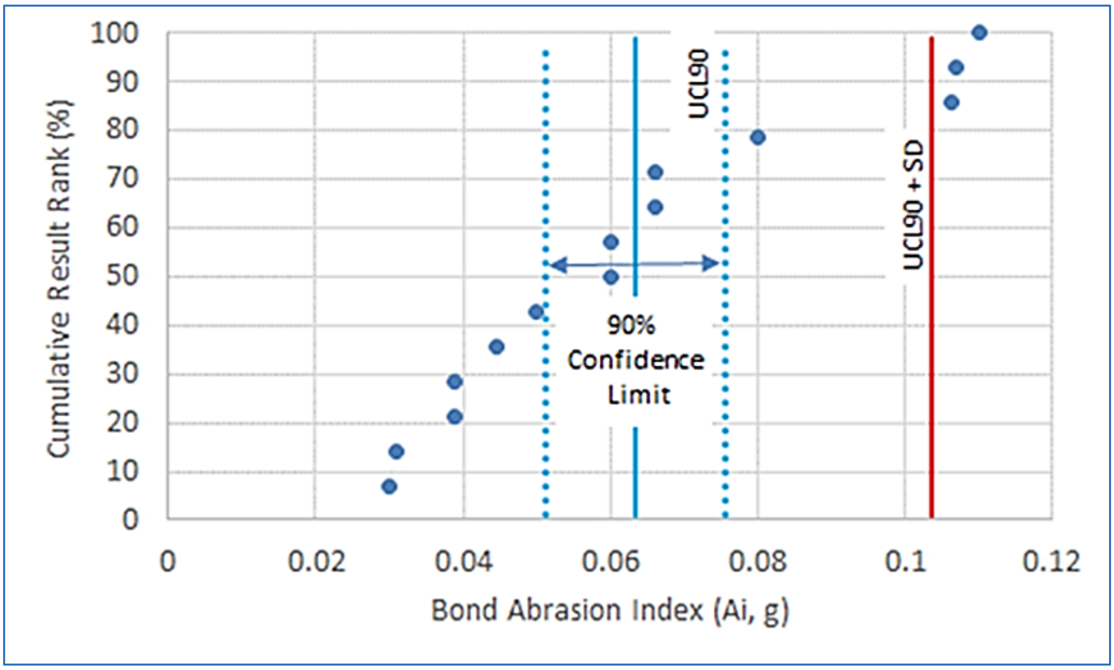


Image courtesy of Amec Foster Wheeler, 2016.

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

If a safe upper level design point was required (similar to an 80th percentile value) then adding the SD to the UCL90 gives a reasonable value, as is also shown on Figure 13.6.

Using confidence limits is a method preferred by Amec Foster Wheeler for estimating a safe average value from small and highly variable data sets because it is much more realistic than simply taking the arithmetic average. In the case of the Ai value, the UCL90 value itself has been chosen as the design point because Ai is a driver of annual operating cost rather than capital equipment selection. In the case of the Axb value and the CWI value a design point has been selected by adding one standard deviation to the UCL90, giving an answer to use in preference to the 80th percentile value. Note that as SAG milling has been rejected as an option, the Axb value has not actually been used in design.

13.6.2 Phase 6 XPS Flotation Testing

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

Figure 13.7 Drill Collars for Phase 6 Flotation Test Composite Samples

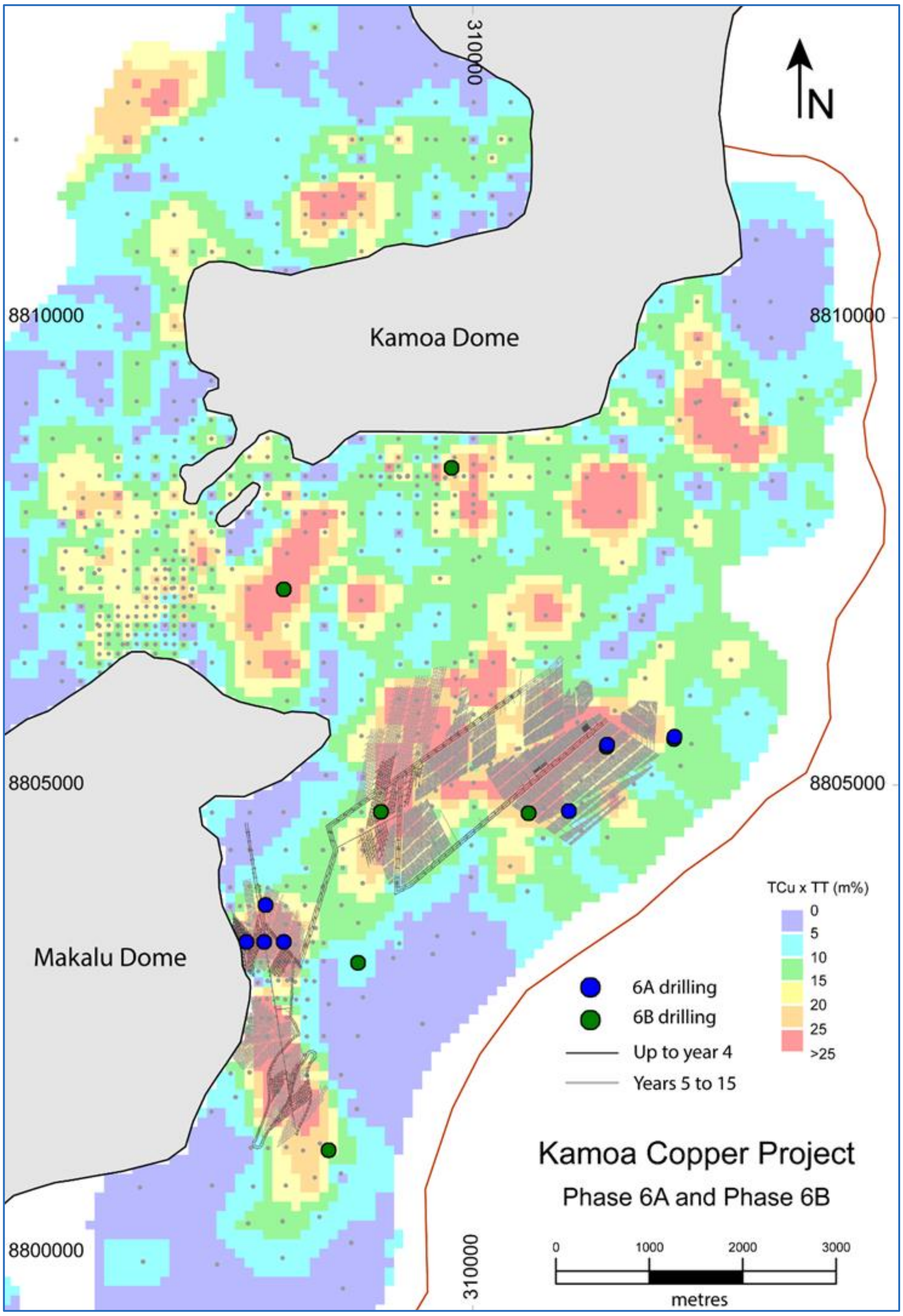


Figure provided by Ivanhoe, 2016.

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

Table 13.13 Phase 6 Flotation Test Composites

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

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

Figure 13.8 Copper to Sulphur Ratios in Phase 6 Composites

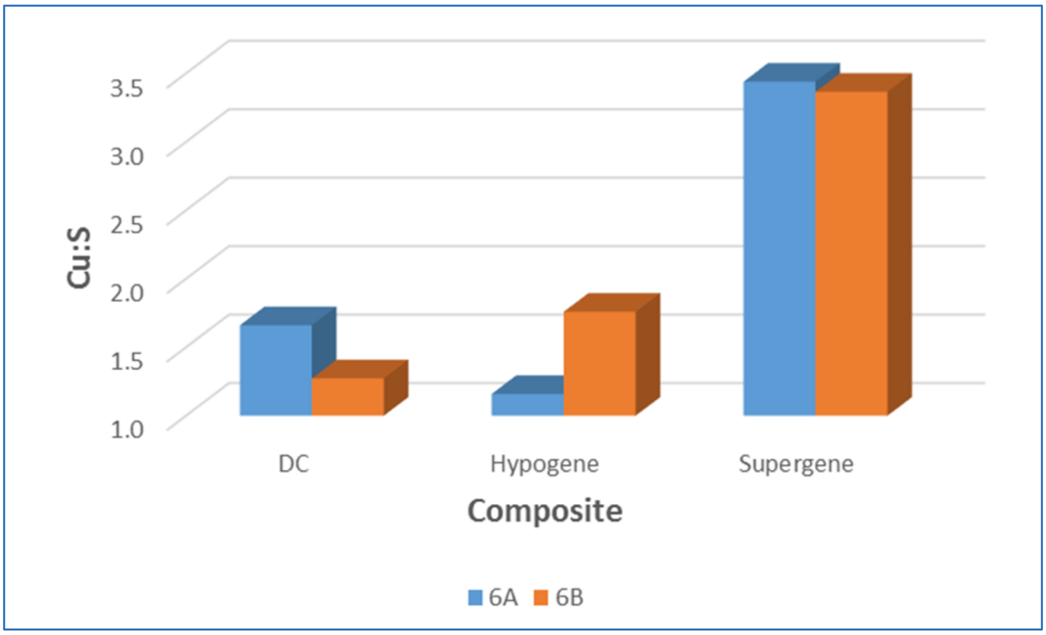


Image courtesy of Amec Foster Wheeler, 2016.

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

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

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

Figure 13.9 QEMScan Copper Mineralogy of Phase 6 Composites

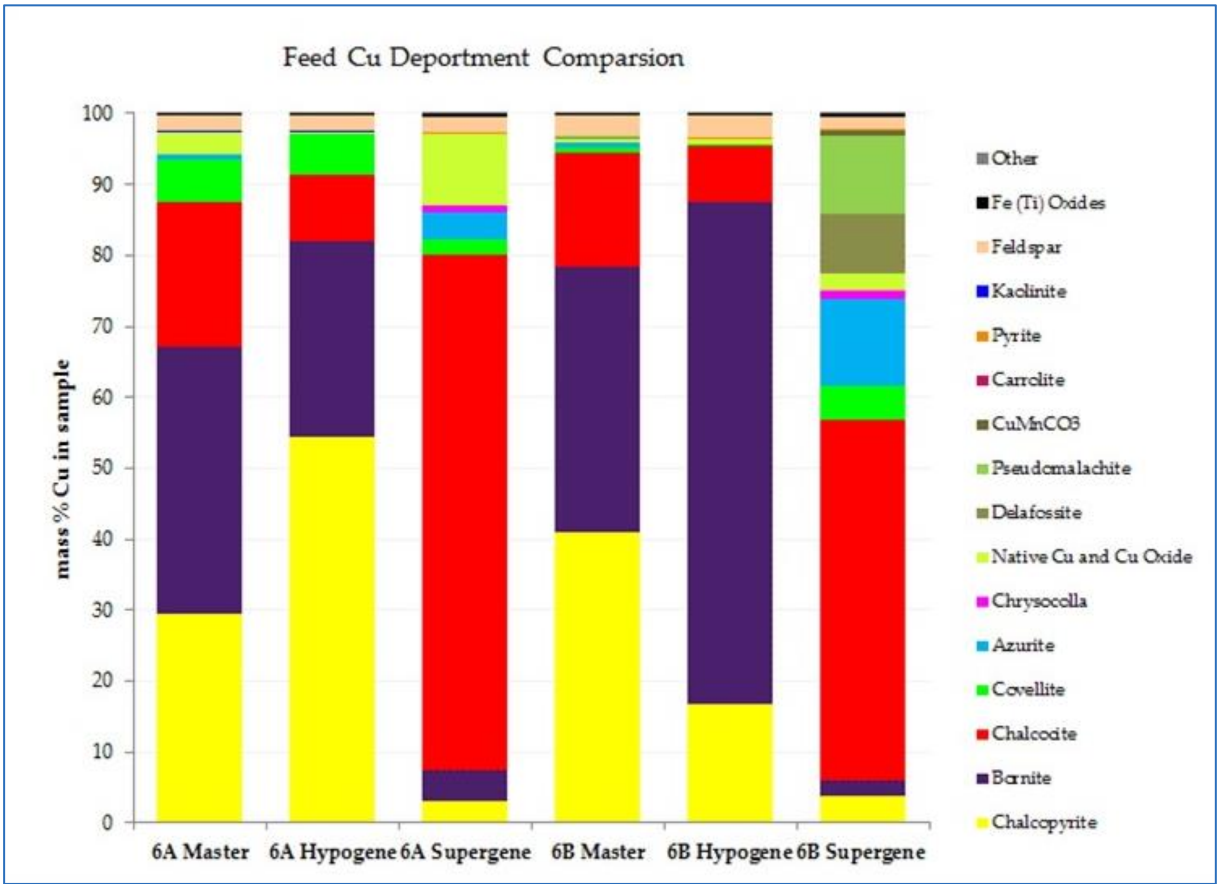


Image courtesy XPS, 2015.

Master sample is an alternative name for DC sample. The DC samples both have a mix of hypogene and supergene. The presence of supergene in the 6B Master sample is best illustrated by the presence of azurite, which is always absent in Kamoa hypogene. The purple band represents bornite which has a relatively high Cu: S ratio. It is the dominance of bornite in the 6B hypogene sample that leads to its anomalous Cu: S ratio. The final flowsheet 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 P_{80}) and fine (10 µm P_{80}) 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 flowsheet. A number of versions of this flowsheet were tested, and the preferred configuration was termed IFS4. The IFS4 flowsheet is shown in Figure 13.10. Each of the six primary Phase 6 composites was tested using this flowsheet and the results are compared in Table 13.14.

Figure 13.10 XPS IFS4 Flowsheet

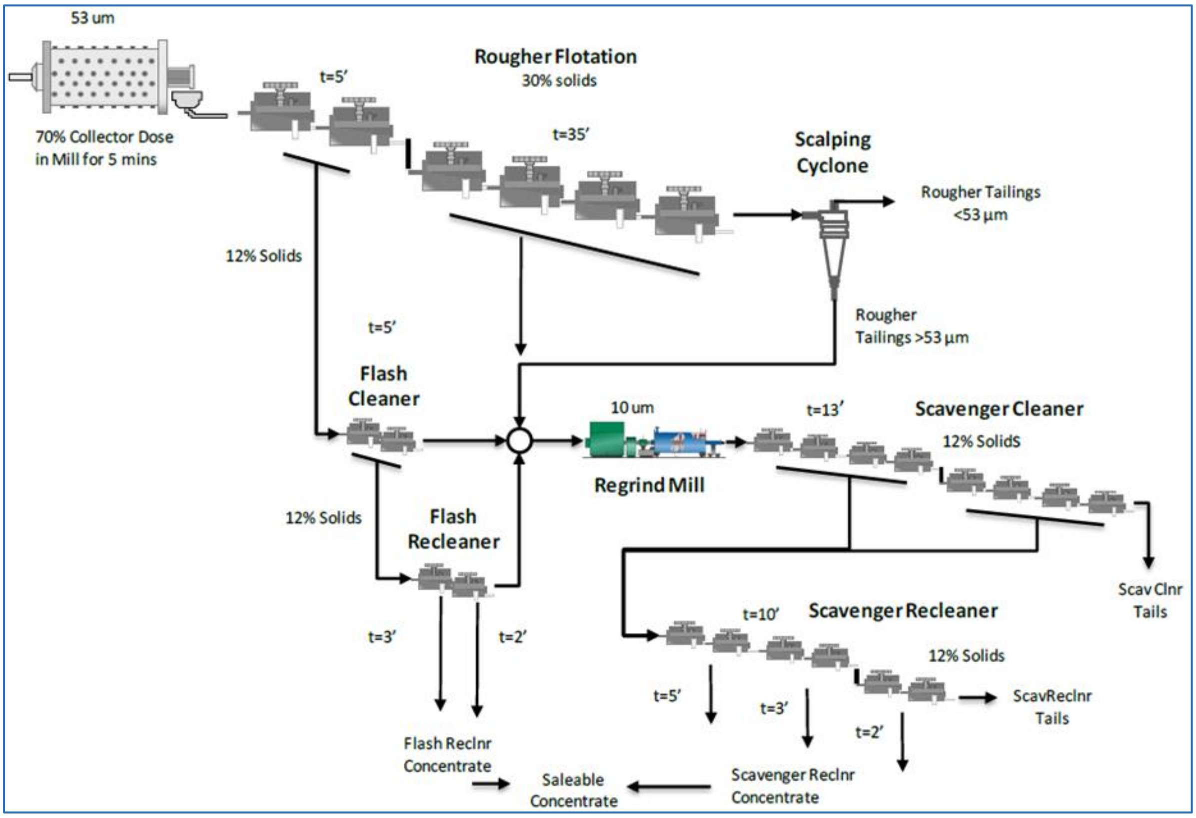


Image courtesy XPS, 2015.

Table 13.14 Flotation Results – IFS4 Circuit

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

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

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

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

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

The flowsheet was simplified to what is termed the IFS4a configuration by removing the 53 µm scalping of rougher tailings. This was done because the practical implications of conducting this scalping step are not well represented in the test method for the following reasons:

1. Scalping would actually be carried out using cyclones which have poor efficiency compared to screens, and more fines would be sent to regrinding and flotation.
2. Scalping using cyclones would also result in a loss of some of the oversize to overflow due to inefficiency.
3. An alternative to cyclone scalping of the tailings would be to grind finer before the roughers.
4. In the IFS4 circuits an average of 45% of the plant feed needs to be ground down to 10 µm with the hypogene and composite samples and about 36% with the supergene samples. These proportions compare with 25% and 21% respectively for non-scalping circuits like IFS4a.
5. These high regrind mass proportions increase even further with the use of cyclones to do the scalping.

The complexity of scalping was removed from the design and testwork was repeated to reflect the recommended PFS circuit. The IFS4a circuit is shown in Figure 13.11. Indicative power requirements for these two circuits at full scale are 29–30 kWh/t for IFS4 and 23–23.5 kWh/t for IFS4a.

Figure 13.11 XPS IFS4a Flowsheet – Basis of the Kamoa 2017 PFS

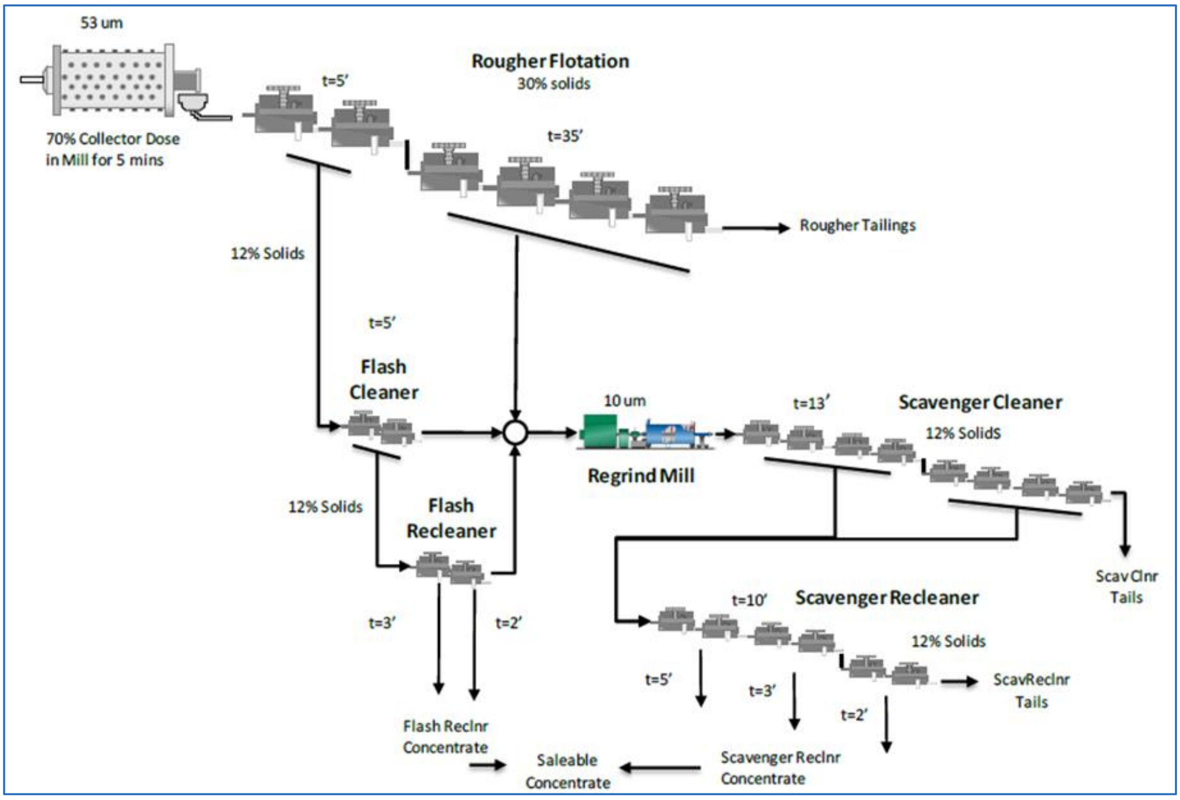


Image courtesy XPS, 2015.

All the tests were repeated with the IFS4a circuit and the results are shown in Table 13.16.

Table 13.16 Flotation Results – IFS4a Circuit

Composite		Final Concentrate					Tail	Feed
		Mass%	% Cu	Rec Cu (%)	%SiO_2	%Fe	% Cu	% Cu
6A	DC	7.80	41.4	86.2	11.1	16.8	0.56	3.74
	90:10 H: S	8.33	37.0	85.4	6.34	22.0	0.58	3.61
	Hypogene	8.48	36.0	86.1	4.00	21.0	0.54	3.54
	Supergene	5.25	53.5	72.3	13.5	13.4	1.14	3.89
6B	DC	8.07	35.4	89.2	9.45	21.3	0.37	3.20
	Hypogene	7.17	35.5	86.9	19.2	13.5	0.41	2.93
	Supergene	6.02	41.2	65.3	19.3	9.65	1.40	3.80

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

On average across the six test samples, the IFS4a flowsheet 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 flowsheet can be selected. However, for the Kamoa 2017 PFS it has been assumed that the benefits of the simpler IFS4a circuit outweigh the losses.

The IFS4a copper concentrate grade and recovery data from Table 13.16 has been plotted in Figure 13.12.

Figure 13.12 Recovery vs Grade Plot for Phase 6 IFS4a Comparative Flotation Tests

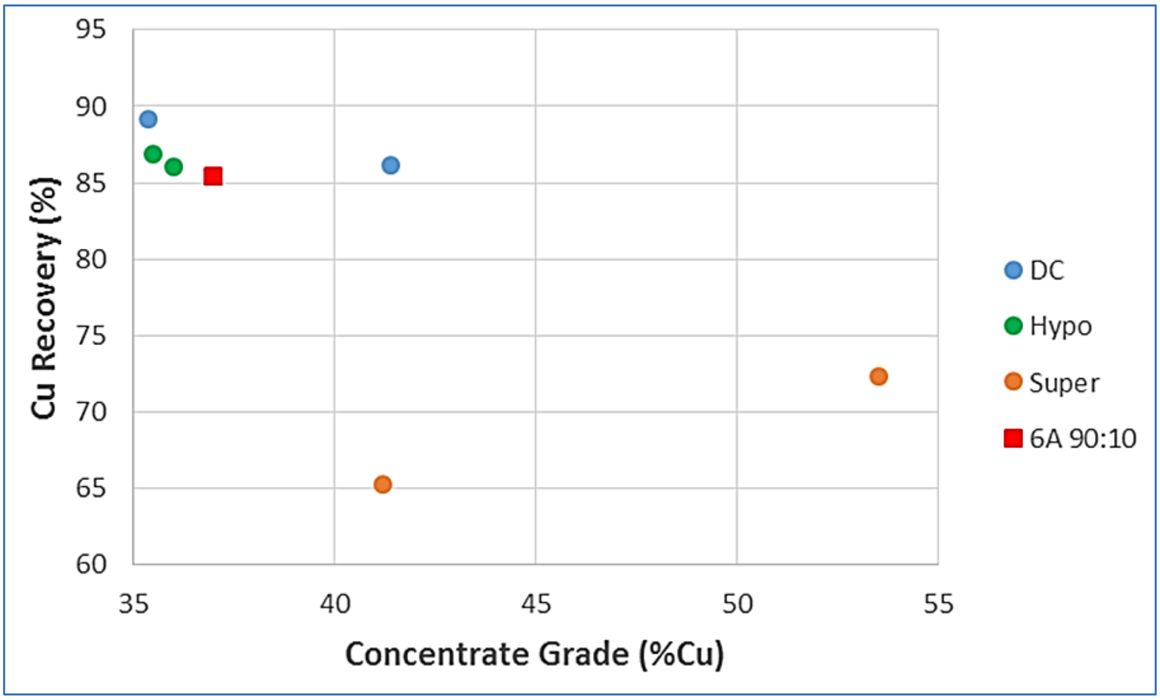


Image courtesy of Amec Foster Wheeler, 2016.

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

13.6.3 Copper Recovery vs Head Grade Model

To allow the prediction of copper recovery in the block model (mine planning) it is usually necessary to develop a model relating copper recovery to head grade. The recovery model from the previous Technical Report is presented in Figure 13.13, together with the performance seen in the Phase 6 IFS4a tests.

Figure 13.13 Old Copper Recovery Model (TR 2013)

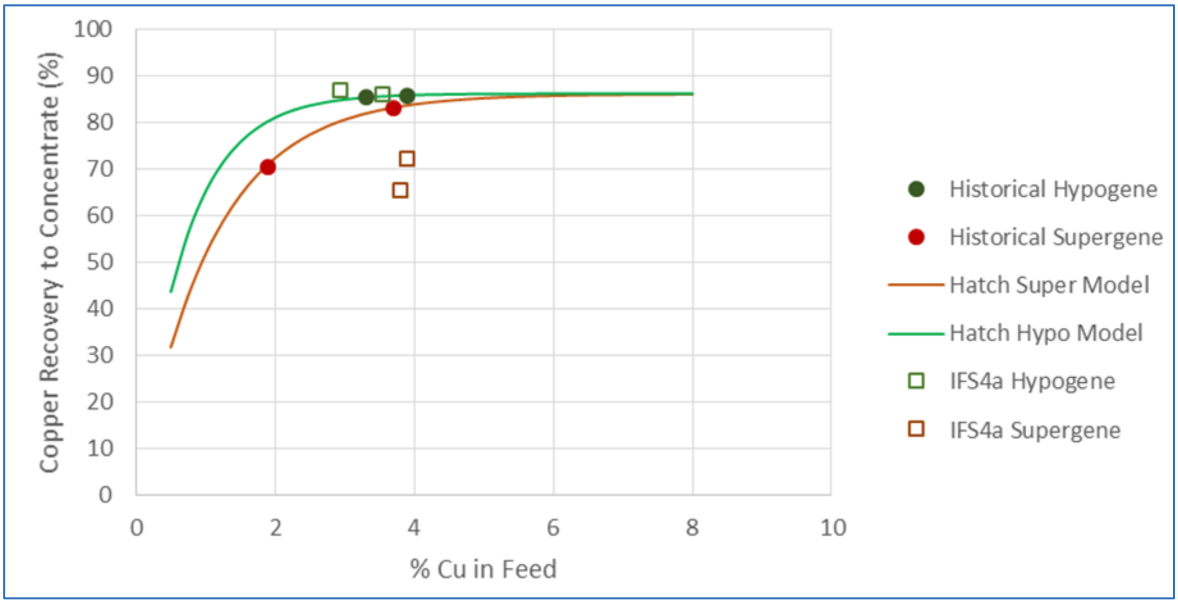


Image courtesy of Amec Foster Wheeler, 2016.

The Phase 6 Hypogene results conform reasonably to the old model, but the supergene response does not. To incorporate the Phase 6 results into the design and planning calculations, improved recovery models are required. In the PEA (2012) a model was developed based on non-floating copper and this has been revived and updated to match the Phase 6 results. As can be seen in Figure 13.14, the new model better represents the Phase 6 results. The new hypogene results were also modelled with less recovery drop-off below 3% Cu.

Figure 13.14 Updated Recovery Models based on PFS Testing

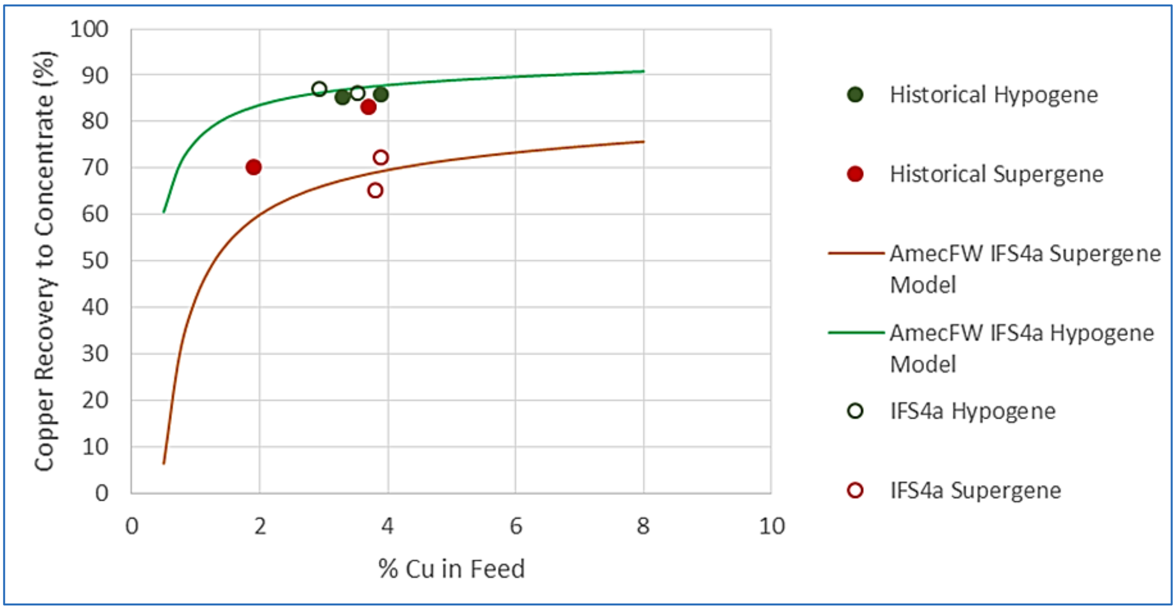


Image courtesy of Amec Foster Wheeler, 2016.

Compared to past models, the new model predicts similar recoveries from hypogene and much lower recoveries from supergene. The lower recoveries for supergene are in line with the test results and are partially the result of high variability in the composition of supergene samples from one test phase to the next. Given that the Kamoa 2017 PFS ore schedule includes the supergene composite samples tested in Phase 6, the modelled recovery reductions are valid.

13.6.3.1 Supergene Recovery Variability

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

The block model contains acid soluble copper (ASCu) information, which allows copper recovery predictions to be made for a subset of the supergene mineralisation type. It is only necessary, at this stage of the project, to modify recovery in mineralised zones where the supergene classification is the result of surface oxidation. It is not necessary if it is classified as supergene due to alteration at depth from fluid originating from the sandstone beneath the mineralised zone. Recovery from all "deep" supergene is calculated using the hypogene recovery formula.

In addition, in some intersections the surface oxidation has not been severe enough to increase the proportion of ASCu above the threshold normally seen in hypogene samples, which is in the range of 5% to 15% (it is thought that the ultra-fine component of the sulphide mineralisation, especially chalcocite, is dissolving during the ASCu determination, but this is yet to be confirmed). Consequently, an alternative recovery calculation is only applied for near surface supergene having greater than 15% of total copper being ASCu.

For the Phase 6 testwork on hypogene and supergene samples, the relationship between floatable copper in the feed (as mineralogically defined using QEMScan Analysis) and copper recovery to concentrate is shown in Figure 13.15.

Figure 13.15 Prediction of Copper Recovery Using Mineralogy

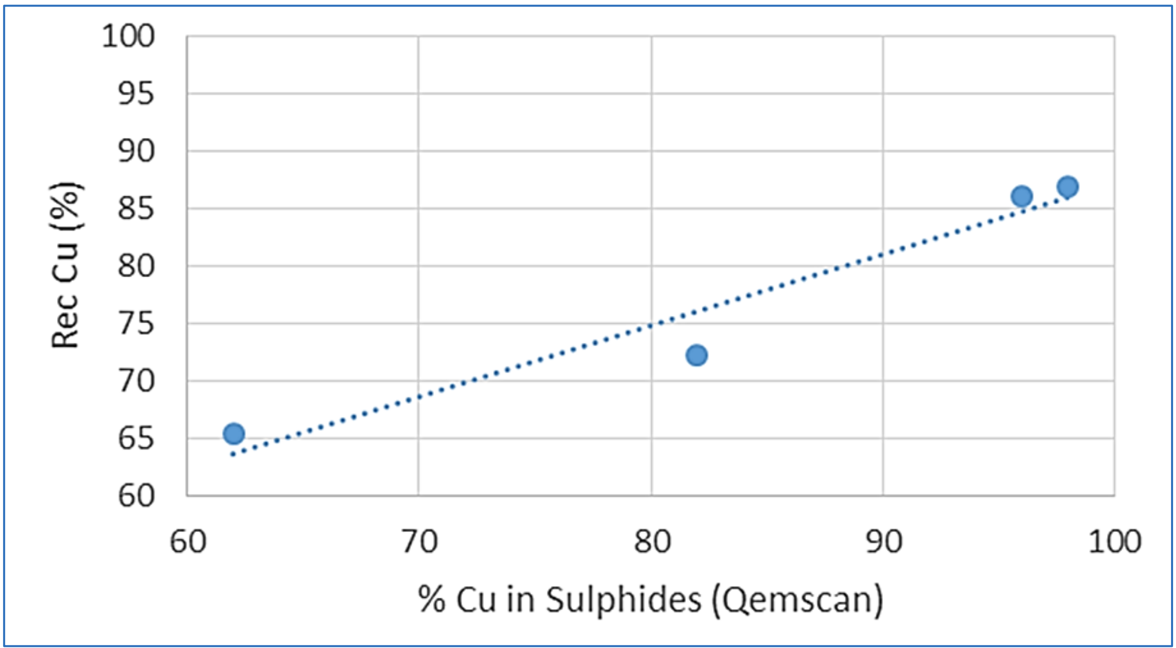


Image courtesy of Amec Foster Wheeler, 2016.

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

13.6.4 Phase 6 Testwork – Signature Plot XPS

A signature plot is used to design 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 flowsheet includes regrinding, it was necessary to truncate the tests ahead of the regrinding stage at each point. The test format is shown in Figure 13.16.

Figure 13.16 Truncated XPS IFS4a Circuit

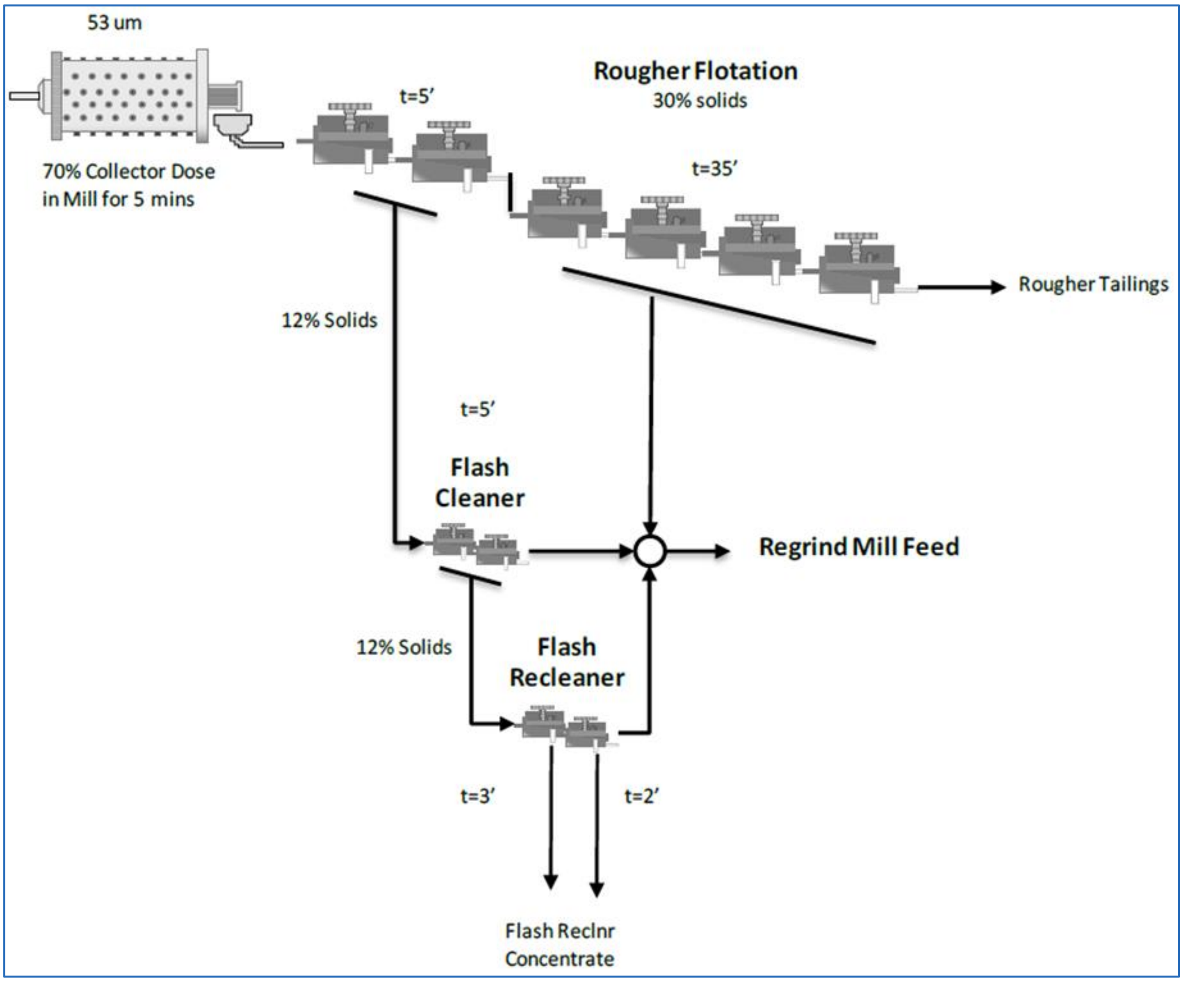


Image courtesy XPS, 2015.

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

Although the rougher feed was ground to a P_{80} of 53 µm, the regrind mill feed was much finer with a P_{80} of 34 µm. The regrind feed contained 56% of material finer than 10 µm and 4% of material coarser than 100 µm. The regrind feed represented 30.8% of the new feed by mass, higher than the 24% of mass estimated for the 6A DC composite. The higher mass is partially driven by the higher feed grade and also increases because the Cu: S ratio is lower.

The IsaMill feed grade was relatively low at 6.6% Cu and contained almost half (47%) of the copper in the test feed. The SG of IsaMill feed was measured at 2.98. Xstrata set the IsaMill feed percentage solids at 41% to avoid viscosity problems potentially associated with a 10 µm regrind target.

The IsaMill feed sample was passed through the M4 IsaMill test unit multiple times, and samples were taken of the product at each pass. The resulting signature plot is shown in Figure 13.17.

Figure 13.17 IsaMill Signature Plot

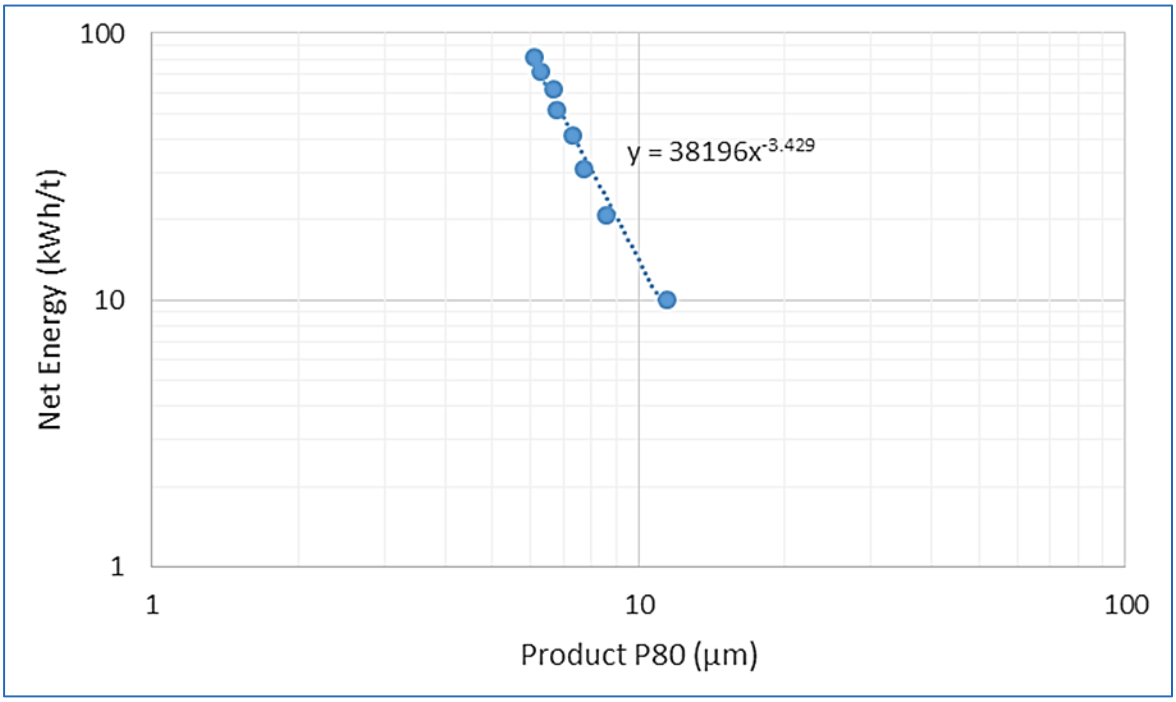


Image courtesy of Amec Foster Wheeler, 2016.

This result is based on the sample tested, and the specific grinding energy requirement for other feeds will be dependent upon the P_{80} of the regrind feed and the mineralogy of the feed. An analysis of the various Phase 6 tests showed that these factors, together with the mass pull to be reground, vary considerably as summarised in Figure 13.18.

Figure 13.18 Phase 6 Regrind Feed Variability

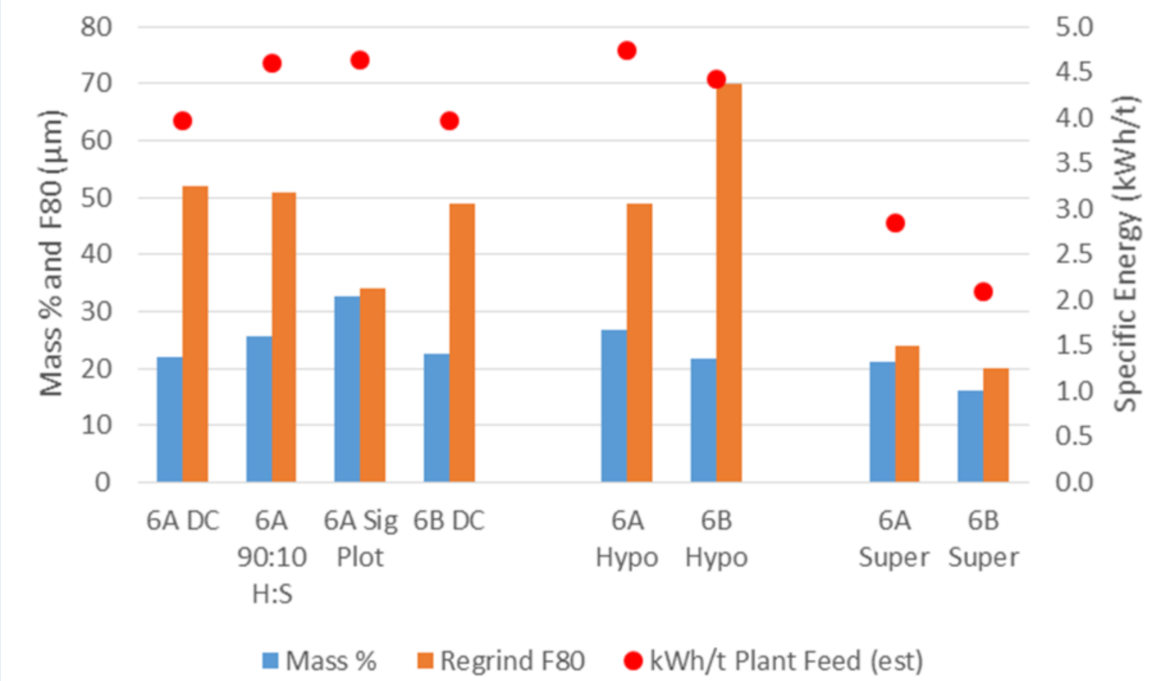


Image courtesy of Amec Foster Wheeler, 2016.

Interestingly, across the four development composites and two hypogene samples the energy per tonne of plant feed is somewhat independent of the test. This is because low mass pulls tend to have coarse particle sizes while high mass pulls are finer. From the Figure 13.18 data, a regrind power selection of 5 kWh per tonne of plant feed should be sufficient to provide regrind capability in the Kamoa 2017 PFS circuit.

The supergene composites only require 3 kWh per tonne of plant feed, but are not planned to be mined or processed in isolation and will not be subjected to overgrinding.

13.6.5 Kamoa Phase 6 Variability Testwork

A program of variability testwork has been planned for Kamoa using the samples indicated in Figure 13.19 together with the Year 0 to 15 PFS mining areas.

Figure 13.19   Planned Phase 6 Variability Samples

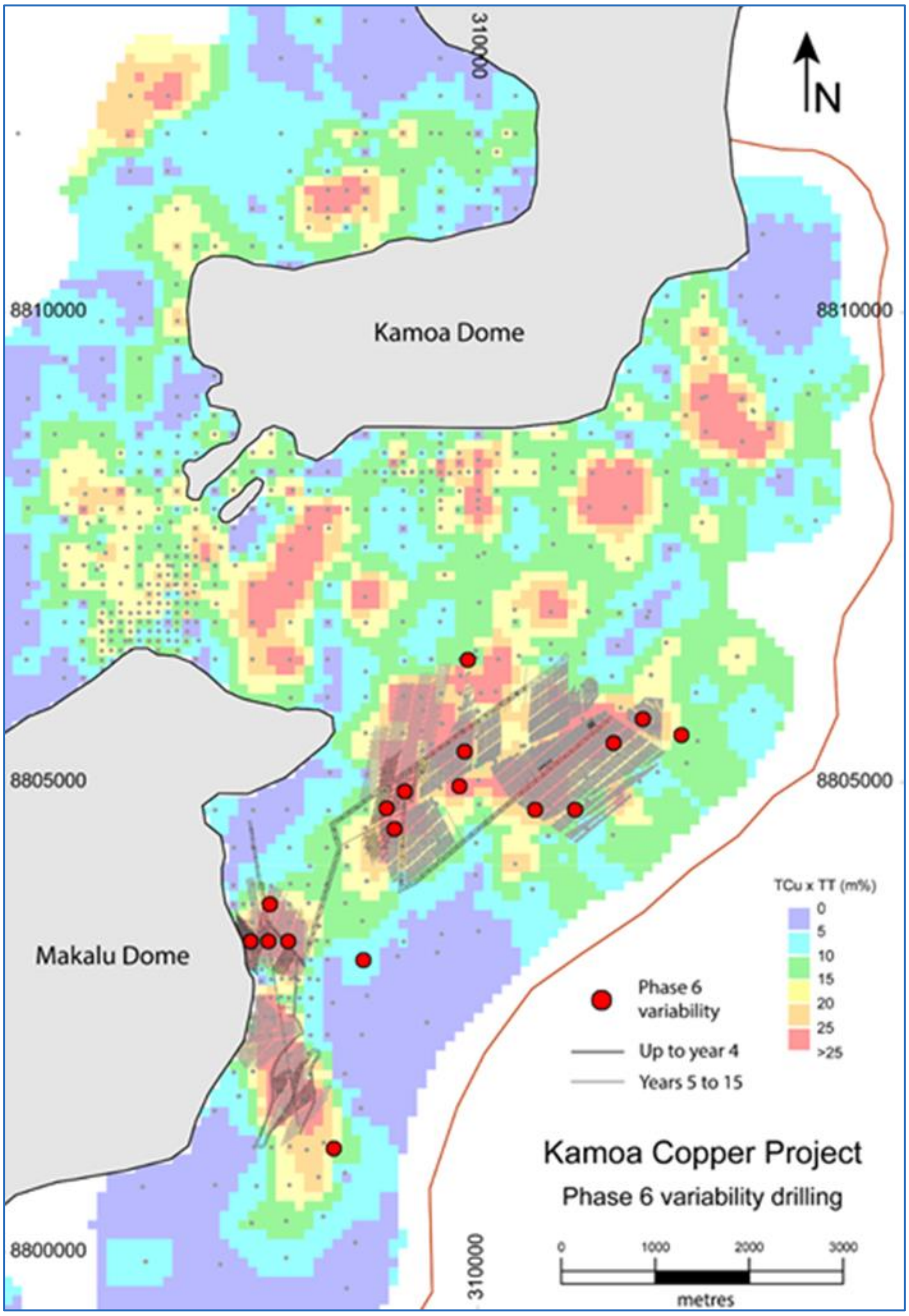


Figure provided by Ivanhoe, 2016.

The variability sample selections provide good spatial representation of plant feed during the proposed Kansoko mine plan period. However, due to shifting project priorities these samples remain in refrigeration ready to be tested in the future.

13.7 Kakula Metallurgical Testwork

Preliminary comminution testing has been carried out on Kakula samples and flotation tests have been conducted using the PFS IFS4A flowsheet and reported below.

13.7.1 Preliminary Metallurgical Testwork Samples

Only three metallurgical PQ holes have been drilled at Kakula for the planned preliminary comminution testwork. Figure 13.20 shows the collar positions of drillholes used for sample collection for flotation and comminution testwork.

Figure 13.20 Metallurgical Drillhole Location Map

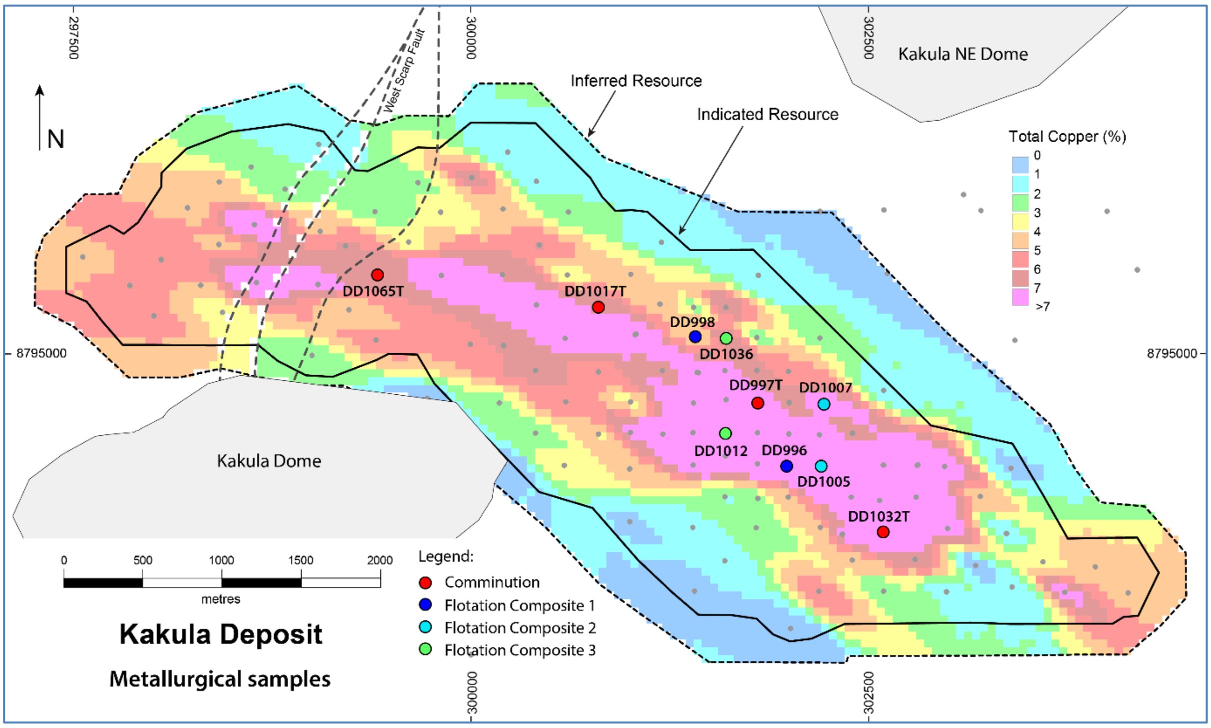


Figure provided by Ivanhoe, 2016.

There are currently no dedicated metallurgical coreholes drilled at Kakula for flotation testwork; however, approximately one quarter of the drill core from the resource holes have been sampled and sent to either Zijin or XPS laboratories for preliminary flotation testwork programs between July and September 2016, as indicated in Figure 13.20 and explained below.

- DD996 and DD998 – Flotation Composite one sent to Zijin.
- DD1005 & DD1007 - Flotation Composite two sent to Zijin.
- DD1012 & DD1036 - Flotation Composite three sent to XPS.

13.7.1.1 Preliminary Comminution Testwork Results

Comminution testing was conducted on individual hole/lithology samples from the Minzones and on composite footwall samples sourced from all four holes. The comminution testing is summarised in Table 13.17.

Table 13.17 Preliminary Comminution Test Results

Sample ID	CWI 75th percentile (kWh/t)	UCS 75th percentile (MPa)	Bond Abrasion Index (g)	BRWI (kWh/t)	BBWI@ 106 µm (kWh/t)	BBWI@ 75 µm (kWh/t)	Head Cu %	Axb	SG
DD1017 SDT	8.8	67	0.020	20.1	17.1	17.1	2.94		
DD1017 SSL	8.8	116	0.010	22.7	15.9	15.4	9.21		
DD1032 SDT	11.6	78	0.020	20.7	16.8	16.9	3.15		
DD1032 SSL	10.1	68	0.010	19.0	15.9	15.4	16.13		
DD1065 SDT	12.0	132	0.010	24.9	18.1	18.8	3.14	22.6	2.84
DD1065 SSL	10.4	205	0.010	24.5	18.0	19.1	6.21		
DD997 SDT	10.8	99	0.050	20.0	16.6	17.6	4.14		
DD997 SSL	10.5	71	0.030	24.1	19.8	19.5	8.95	23.4	2.94
FW SDT	11.0	69	0.060	20.7	17.8	17.9	1.18		
FW SST	11.9	94	0.320	16.1	16.6	17.8	0.78		
Kakula Minzone ave	10.4	105	0.020	22.0	17.3	17.5	6.73	23.0	2.89
Kamoa Phase 6 ave	11.3	144	0.080	19.1	17.0	–	3.51	17–28 (hist)	2.70

Kakula mineralisation is consistent with the Kamoa samples on most measures, with the major exceptions being grade, rod mill work index (BRWI) and Bond abrasion index. The abrasion index and the copper grade are both favourable but the BRWI is significantly worse. The higher BRWI value has resulted in recommendations to reduce the ball mill feed topsize as much as is practical so as to minimise or avoid scatting of the primary ball mill.

13.7.1.2 Preliminary Metallurgical Testwork Results

The initial Kakula testwork was conducted at Zijin laboratories in China using sample from drill cores DD996 and DD998. The core was crushed and split into two halves. The other half was sent to XPS laboratory in Canada to perform a confirmatory flotation tests. The scope of work for both laboratories included head analysis, grind calibration, and duplicate float tests on each of the core as well as a composite of the two cores using the IFS4A PFS flowsheet developed at XPS.

The head assay determined by Zijin on Kakula composite 1 is summarised in Table 13.18.

Table 13.18 Zijin Results of Head Assay (%)

Sample	%Cu	%S	%Fe	%CaO	$\%Al_2O_3$	%MgO	$\%SiO_2$
Composite 1	4.08	1.2	5.07	2.19	12.66	4.71	55.48

Note that the two laboratories reported many more metals and compounds than shown in Table 13.18 in their comprehensive head grade analyses. The XPS information measured on the same sample was consistent, indicating sample and assay consistency between the laboratories.

The Composite 1 flotation test results summary from Zijin in Table 13.19, showed that the laboratory achieved a copper recovery of 86% and produced a concentrate with 53% copper using PFS IFS4a circuit as shown in Figure 13.1. The results also indicate that material from the Kakula and Kamoa Kansoko zones could be processed through the same concentrator plant, which could yield significant operational and economic efficiencies.

Table 13.19 Summary of Flotation Results – Composite 1

Sample	Concentrate Mass (%)	Cu (%)	Rec Cu (%)	$\%SiO_2$	%Fe
Composite 1	6.6	52.8	86.1	14.4	4.4

However, XPS testwork on the same Composite 1 sample (drill core DD 996, DD 998) reported lower product grades and recoveries than Zijin. Errors in results between individual and composite tests and significantly lower mass pulls than Zijin testwork proves these results to be flawed and justifies not using them in design. Repeat tests on more representative feed samples were recommended and actioned.

Additional samples were prepared and sent for testing in order to address higher feed grade flotation performance and to get results that were comparable between the two laboratories, these were obtained from cores:

- DD1005 and DD1007 - Flotation Composite two sent to Zijin.
- DD1012 and DD1036 - Flotation Composite three sent to XPS.

The Zijin testwork results for Composite 2 (samples DD1005 and DD1007) are compared with the Composite 3 (samples DD1012 and DD1036) results from XPS.

Head assays of the two composites are presented in Table 13.20.

Table 13.20 Triplicate Averaged Head Grades of the High-grade Kakula Composites

Sample	%Cu	%S	%Fe	%CaO	%Al_2O_3	%MgO	%SiO_2
Composite 1	8.19	2.00	4.92	0.96	13.24	3.47	52.82
Composite 2	8.12	1.95	4.97	0.86	13.27	3.76	52.34

From a grade perspective the two composites can be considered identical. Unfortunately, it is not possible to compare the mineralogy of the two samples, as Composite 2 was not analysed by Zijin.

The standard IFS4A flowsheet configuration was maintained by both laboratories for testing. The summary results from the two laboratories are compared in Table 13.21.

Table 13.21 Summary of Standard Procedure Flotation Results

Sample	Concentrate Mass%	% Cu	Rec Cu (%)	%SiO_2	%Fe	As
Composite 2	12.3	55.6	85.0	13.7	3.8	0.012
Composite 3	12.5	56.0	87.8	14.4	4.2	<0.01

These test results are consistent with each other and were achieved with head grades more representative of the Kakula resource than Composite 1. The IFS4A flowsheet is confirmed as being suitable for treating the Kakula mineralisation. It should be noted that Kakula plant design accommodates the higher mass pulls when treating Kakula mineralisation.

13.7.1.3 Optimisation testing

Both Zijin and XPS conducted optimisations tests which were variations on IFS4A. The best result, achieved by Zijin, is summarised in Table 13.22.

Table 13.22 Summary of Optimised Flotation Results

Sample	Concentrate Mass%	% Cu	Rec Cu (%)	$\%SiO_2$	%Fe	As
Zijin Composite 2	11.9	60.5	87.9	15.4	4.1	0.008

The repeatability of the IFS4A method was confirmed by this work and it was also evident that the results are close to optimal for the new composites tested. Note that the Kakula samples required greater collector addition rates than Kansoko samples in line with the increased proportions of sulphide minerals at Kakula.

13.8 Process Mineralogy

The Kamoa and Kakula deposit mineralogy is discussed in two parts, firstly being the previously defined Kamoa deposit and then the newer Kakula deposit, highlighting the mineralogical changes. Note that the Kakula analysis has been performed on the low-grade composite 1. However, the similarity in the final concentrate grades achieved with both low and high-grade Kakula composites suggests that the mineralogical analysis presented here is valid across Kakula regardless of grade.

13.8.1 Kamoa Mineralogy

The Kamoa copper sulphide mineralisation exists in two basic modes regardless of copper sulphide mineral. Coarse copper sulphides, some in the centimetre size range, are clearly visible in the core. Many intermediate sized copper mineral grains are usually visible but any that are clearly distinguishable can be considered coarse. The second mode of occurrence is a pervasive "fog" of ultrafine copper sulphides throughout the matrix.

In the image below (Figure 13.21) can be seen a 2 cm wide white clast within the grey diamictite matrix, against which chalcopyrite has "mantled" during the sulphide deposition phase. In the surrounding rock matrix there are smaller mantled clasts and visible blebs of chalcopyrite (and other sulphides). What cannot be seen in the photograph is the dispersion of 1 to 10 µm (0.0001 to 0.001 cm) copper sulphides present throughout the grey matrix.

Figure 13.21 Typical Kamoa Hypogene Mineralisation in Diamictite

Figure Courtesy Amec Foster Wheeler, 2011.

QEMScan, an automated particle analysis system, has been used to reveal the fine mineralogical detail of Kamoa samples. Two rougher flotation tests were conducted on the 6A development composite by XPS, in which six concentrates were collected sequentially after grinding the samples to P_{80} 53 µm and 38 µm respectively. The QEMScan analysis was used to derive the proportion of liberated copper in each of the concentrates, and the results are summarised in Figure 13.22.

Figure 13.22 Copper Sulphide Liberation in Rougher Flotation

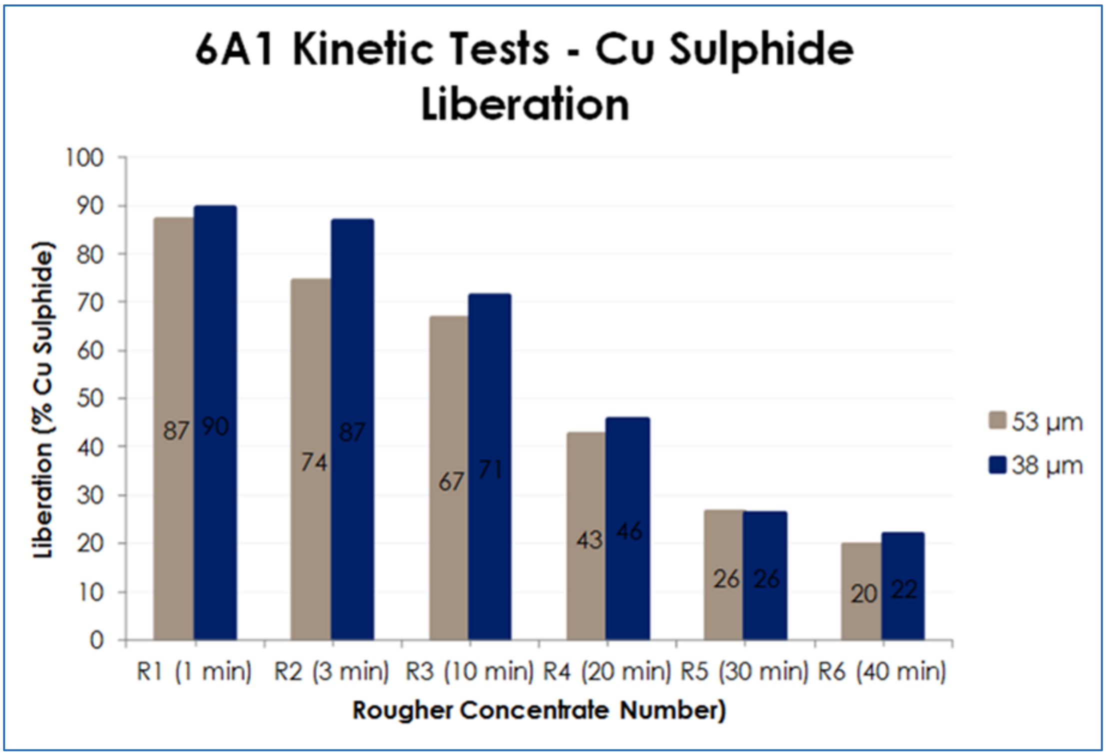


Image courtesy XPS, 2015.

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

Copper sulphide morphology in all Kamoa and Kansoko samples is consistent in that the minerals are always present as both very coarse and very fine grains. The large proportion of copper in fine sulphides is the reason for the strong liberation effect of grinding (measured using QEMScan, XPS Laboratory) as shown in Figure 13.23.

Figure 13.23 Phase 6 Hypogene Composite Liberation Analysis

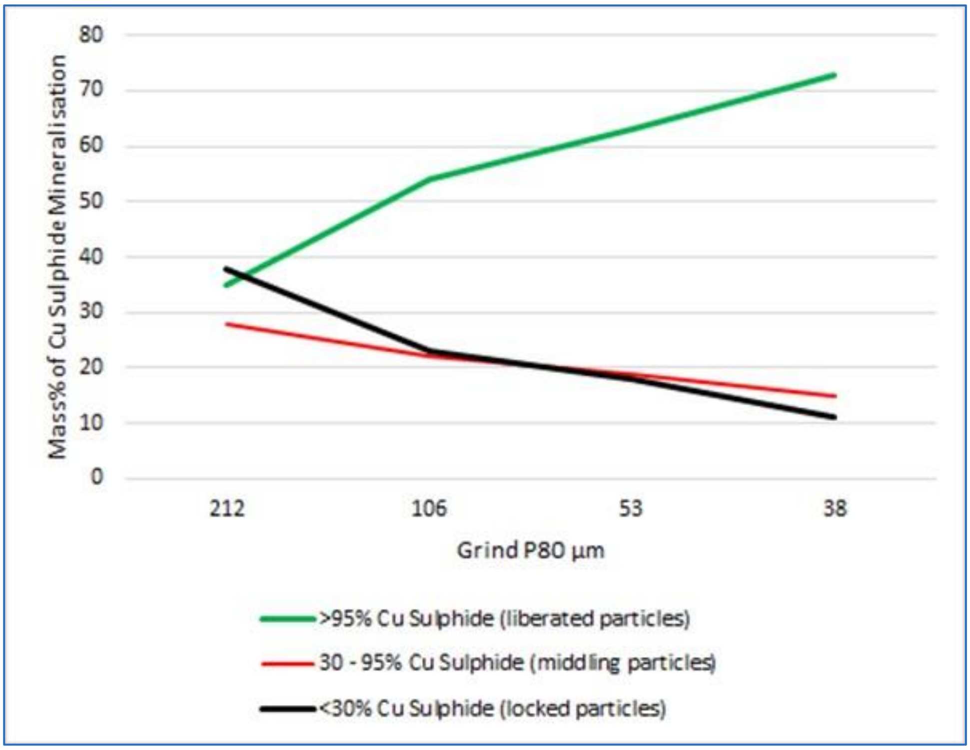


Image courtesy Amec Foster Wheeler, 2018: from XPS data, 2014.

At the fine grind P_{80} of 38 µm, 27% of the copper sulphides remain unliberated. Almost half of these are in the very poor grade "locked" class and are generally unavailable to recover in flotation. If locked particles are recovered they rarely survive the cleaning process and are rejected to tails at some point in the flowsheet.

QEMScan also generates particle mineral maps and is able to group both minerals and particles to assist in visual examination. Figure 13.24 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 13.24 Combined Copper Sulphides Liberation Map – Rougher Concentrates R3 to R6

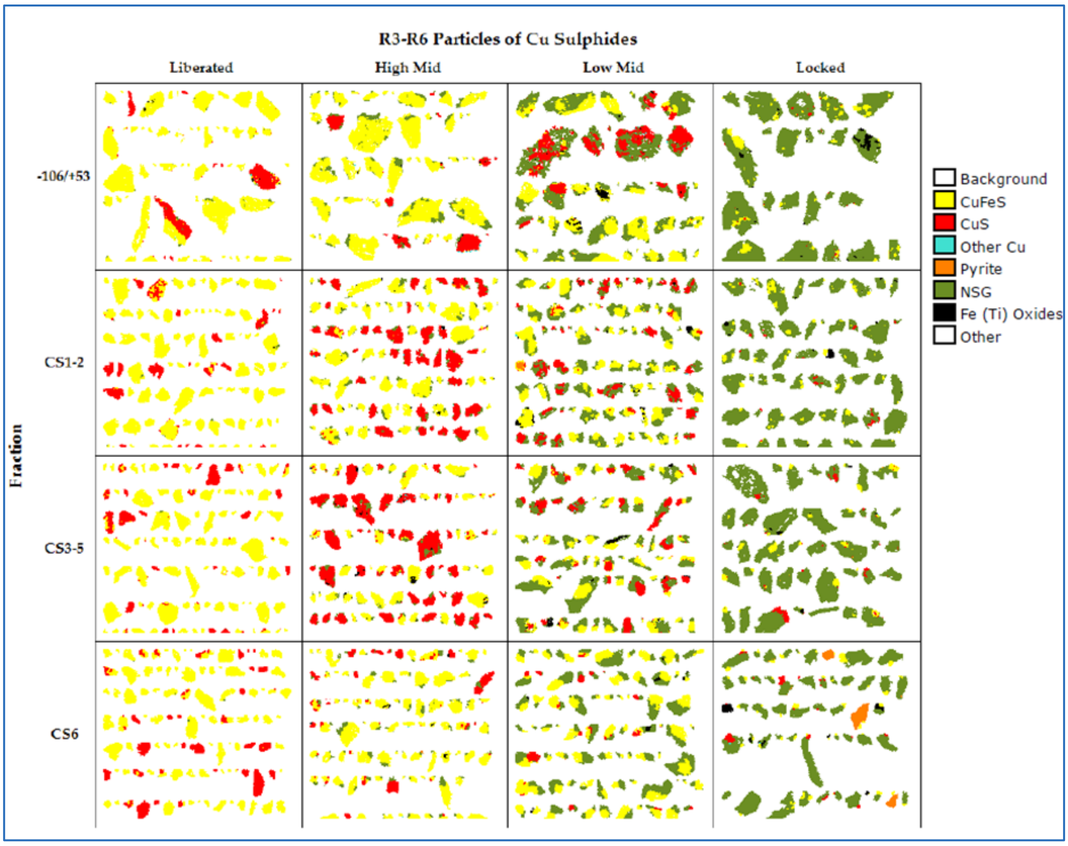


Image courtesy XPS, 2015.

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

All particles in Figure 13.24 above have been floated or transported to the concentrate by entrainment with the froth water. All that is needed for a particle to float is a small exposure of copper sulphide at the surface and the "low Mid" and "Locked" particles in the image shows that this is generally the case.

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

The fine sulphides also mean that regardless of the rougher flotation size it is necessary to regrind middlings material to ultra-fine sizes to achieve low silicate levels in final concentrates. Testing has shown the concentrate quality to be sensitive to regrind P_{80}, with 15 µm producing poor concentrates and 10 µm generally producing acceptable concentrates.

Another notable aspect of Figure 13.24 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 13.25.

Figure 13.25 Combined Copper Sulphides Liberation Map – Rougher Tails

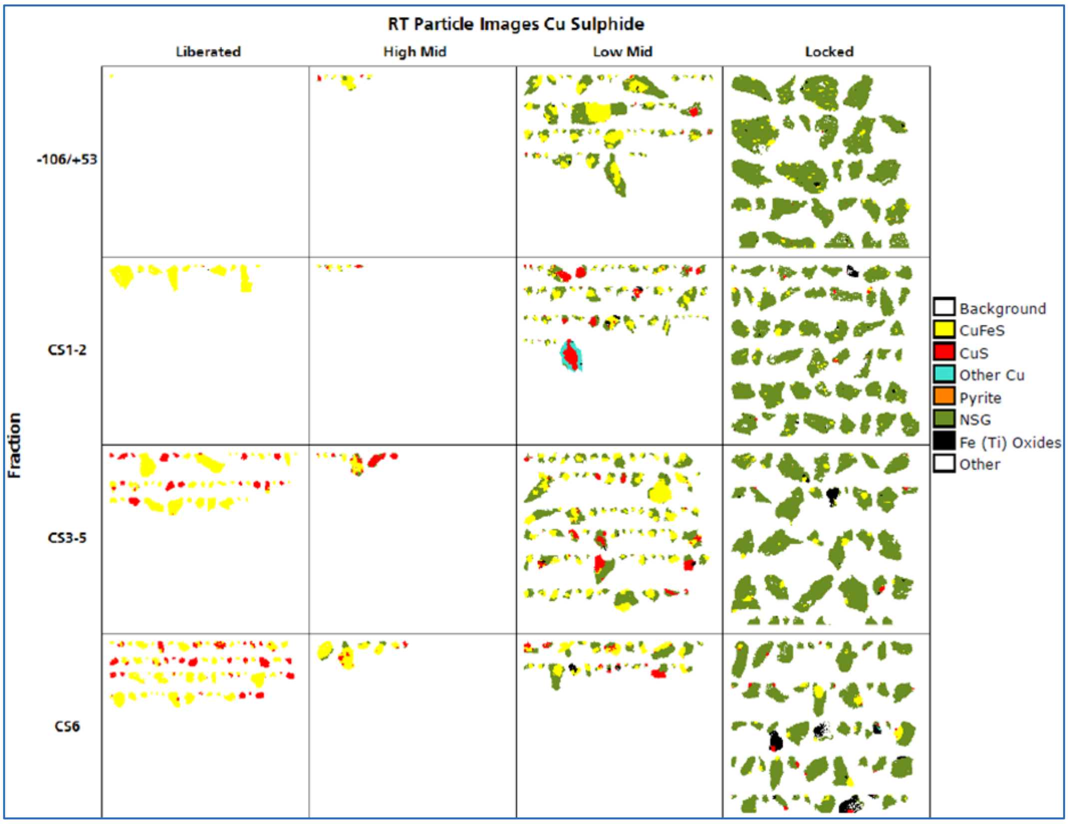


Image courtesy XPS, 2015.

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 13.26, regardless of the size fraction, the lost copper sulphides are in phases that have average grain sizes of less than 10 µm.

Figure 13.26 Copper Sulphide Phase Size in Rougher Tailings

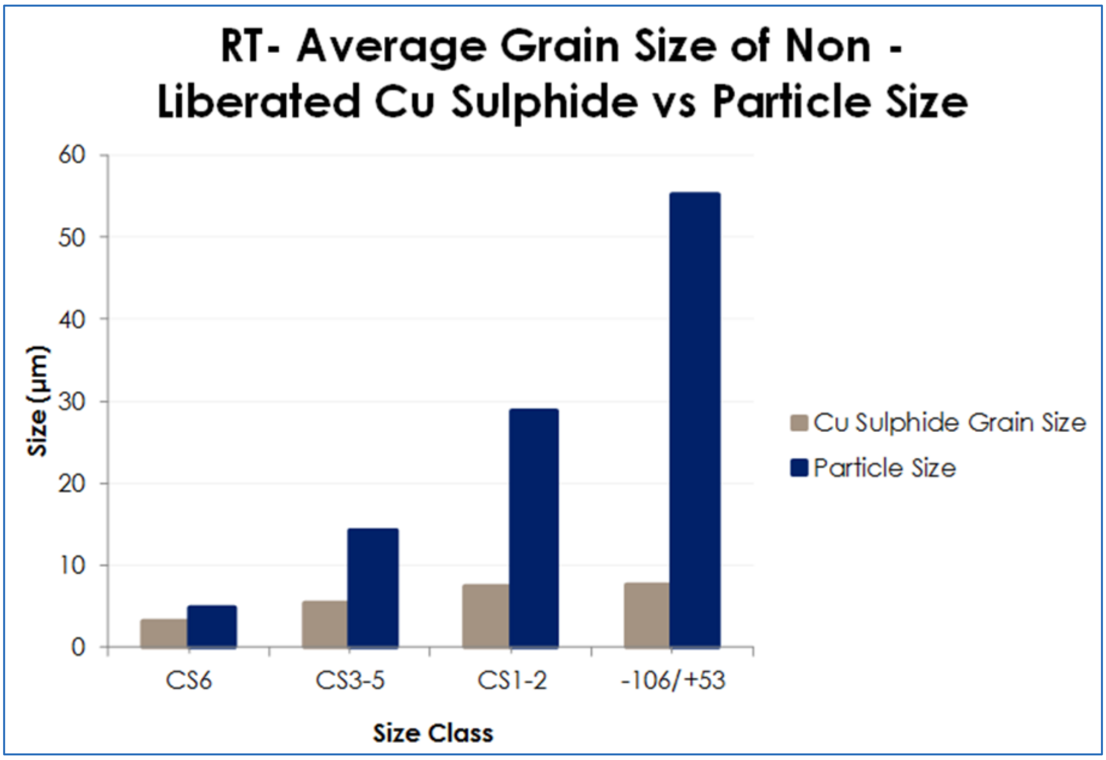


Image courtesy XPS, 2015.

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

13.8.2 Kakula Mineralogy

Two drill cores DD996 and DD998 were combined to make up a Composite 1 for the initial Kakula testwork program. The bulk modal mineralogy results of the composite sample indicate that the dominant base metal sulphides are chalcocite (3.6%) followed by bornite (1.1%), and the gangue minerals are mainly muscovite (49%) and quartz (25%), with minor amounts of biotite, chlorite, K-feldspar, carbonates, iron oxide and trace amounts of native copper, pyrite, apatite, rutile, and kaolinite.

Copper deportment of the Kakula composite 1 sample is chalcocite dominant followed by bornite and digenite (Cu_9S_5, 78% Cu content). The Kakula result is compared in Figure 13.27 to the mineralogically different Kamoa Phase 6 development composite sample, which was bornite-rich and also contained significant amounts of chalcopyrite and some chalcocite.

Figure 13.27 Cu Deportment Comparison Between Kamoa 6ADC and Kakula Composite 1

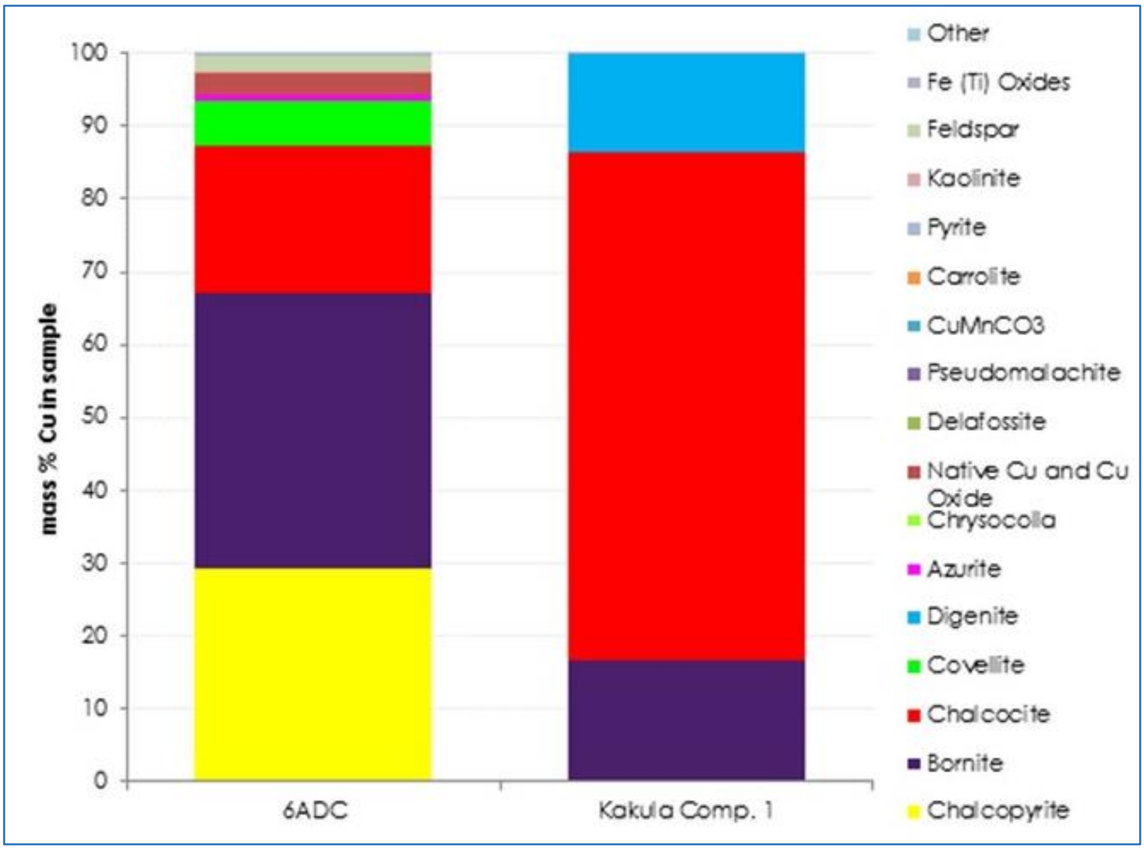


Figure provided by Ivanhoe, 2016.

Diginite is very similar chemically and in its recovery response to chalcocite. For simplicity, its mineralogical measurements have been combined with chalcocite in the analysis and discussion below. When chalcocite is used it refers to chalcocite plus digenite.

The grain size of the copper sulphides in Kakula composite 1 (Kakula Comp, average 33 µm) is coarser than the Kamoa Phase 6 development composite (6ADC, average 20 µm) as shown in Figure 13.28. Bornite was comparatively finer than chalcocite, digenite and chalcopyrite for both samples.

Figure 13.28 Combined Cu Sulphide Grain Size Distribution Comparison Between Kamoa 6ADC and Kakula Composite

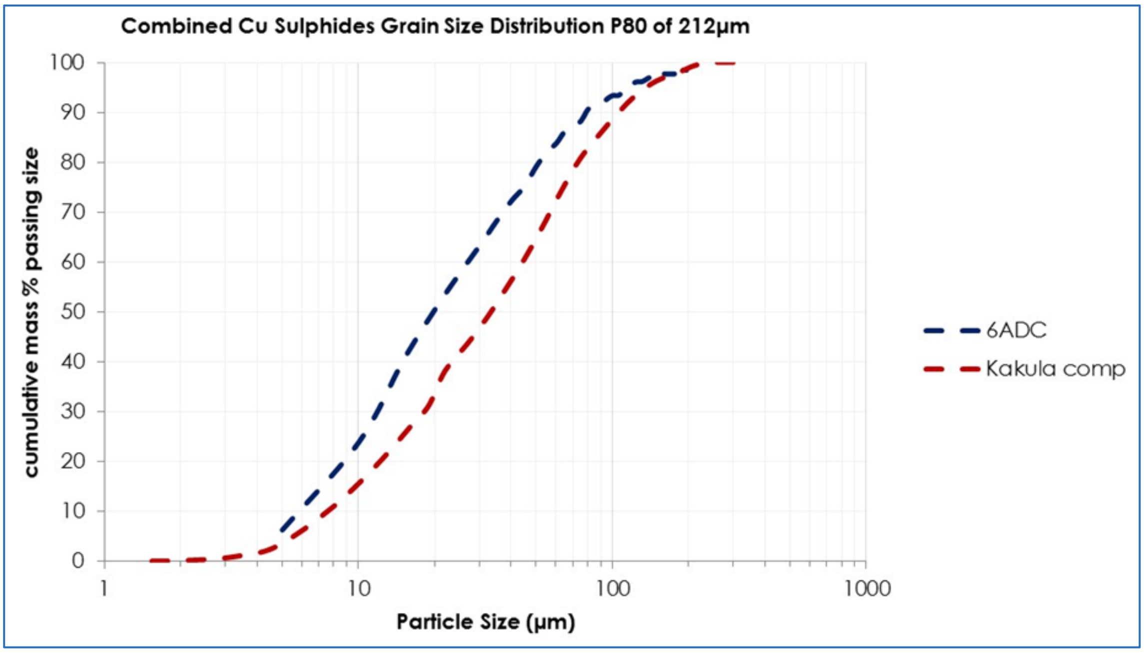


Figure provided by Ivanhoe, 2016.

In Figure 13.29 the copper mineral liberation outcomes for Kakula and Kamoa samples, each after a 212 µm P_{80} grind, are compared. The total of the "liberated plus free" classes is effectively equal for each sample at about 45%, but much more mineral is in the "free" class for Kakula. There are major differences at the "locked" end of the comparison, with Kakula having about half the locked Cu of the Kamoa sample. Most of the unliberated copper sulphides in both samples were locked in, or attached to, non-sulphide gangue.

Figure 13.29 Combined Cu Sulphide Liberation Comparison Between Kamoa 6ADC and Kakula Composite

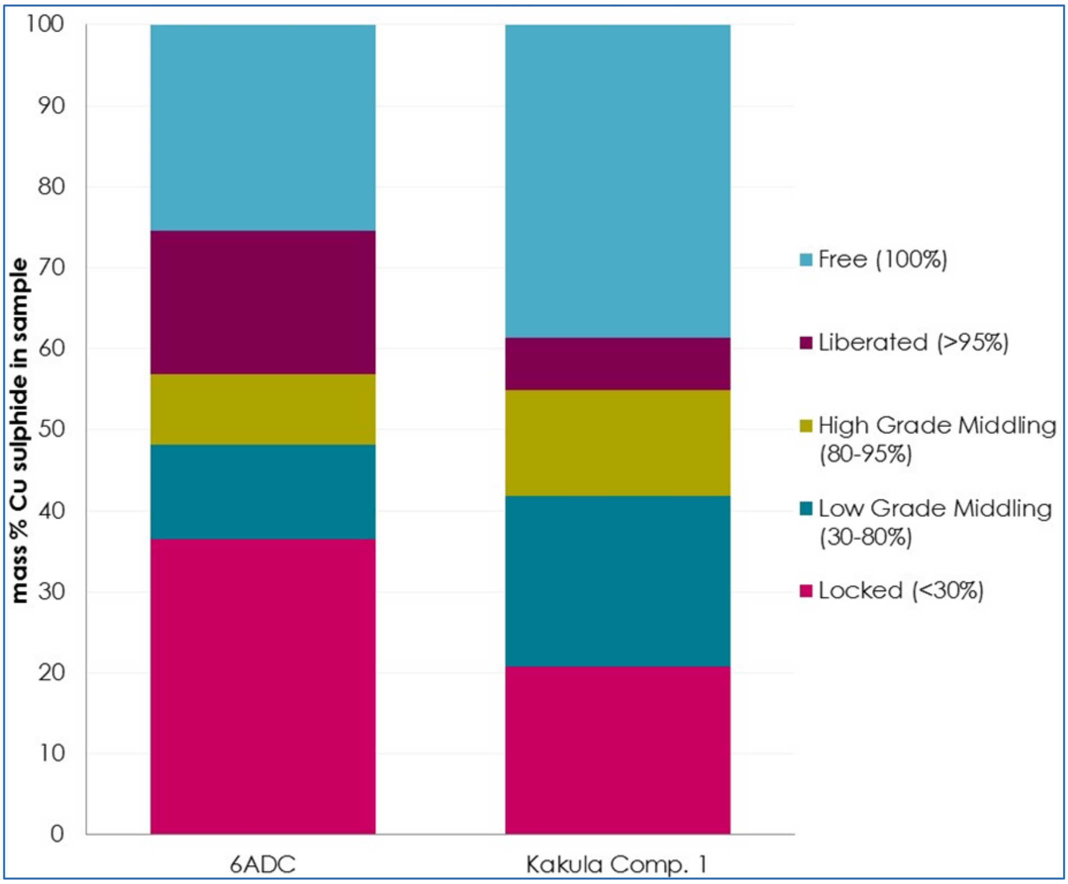


Figure provided by Ivanhoe, 2016.

Chalcocite is a high-tenor mineral that is opaque and dark-grey to black with a metallic lustre. Owing to its very high percentage of contained copper by weight and its capacity to produce a clean, high-grade concentrate, chalcocite is an asset as a dominant copper mineral. Unlike Kamoa, the Kakula deposit has very low bornite, chalcopyrite or other sulphide minerals as seen in Figure 13.30.

Figure 13.30 Comparison of Cu: S between Kamoa and Kakula Mineralisation

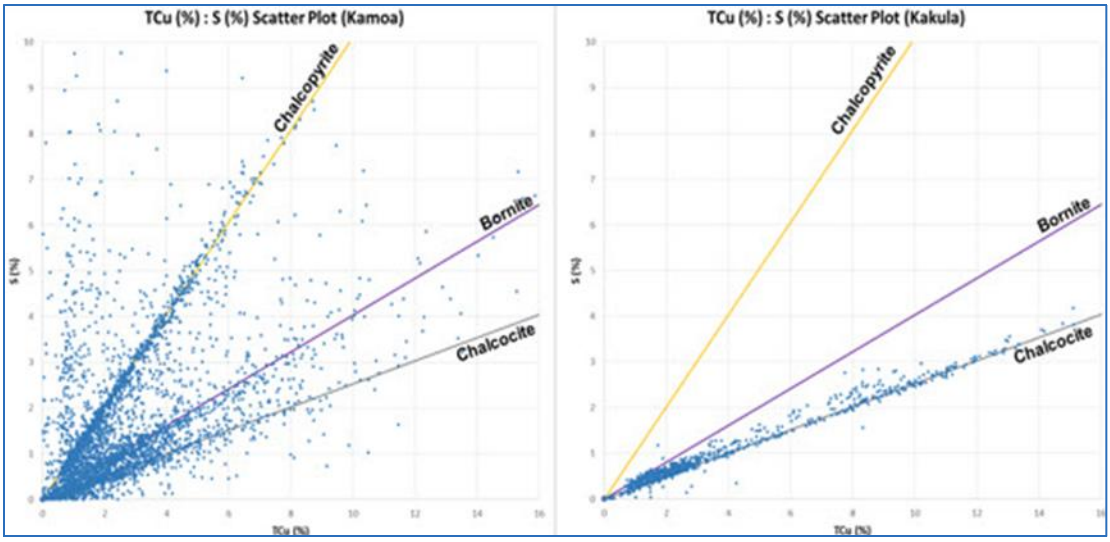


Figure by Ivanhoe, 2016.

This suggests that the Kakula deposit will be easier to treat than Kamoa, and it will be easier to maintain a consistent copper concentrate grade. Mineralogical characteristics are constant across the deposit to date, and there is no indication that they will change significantly. The relatively coarse copper sulphide grain size, the simple mineralogy and the lack of arsenic in feed means that Kakula will generate more valuable concentrates than Kamoa or Kansoko.

13.9 Comments on Section 13

In the opinion of the Amec Foster Wheeler QP the metallurgical testwork conducted for the Kamoa and Kakula deposits is sufficient for PFS level process design. The comminution characteristics are well established and have consistency across the various testing phases and across the three prospective mining areas. The flotation characteristics are consistent (despite highly variable mineralogy), well understood and explainable in terms of the process mineralogy. The samples tested in Phase 6 and in the Preliminary Kakula phase reasonably represent the material to be mined and processed according to the Kakula PEA mine schedule.

The project mineralised zones do not contain deleterious elements often found in copper concentrates, such as arsenic and fluorine and Kakula is especially low in Arsenic. As a result, the flotation testwork has consistently generated concentrates that are free of penalty elements.

The pervasive presence of ultrafine copper sulphides in all Kamoa samples leads to strong recovery of silica through attachment with these sulphides. This, in turn, has led to high rougher mass pull rates and silica rejection issues in final concentrate production, which is mitigated to a large degree by 10 µm regrinding of middling streams. The most recent testwork, at two independent laboratories, has consistently achieved silica levels in the range 14 to 15% $SiO2$ and has provided confidence that this level of silica rejection, at a minimum, will be achievable in operations.

The prediction of copper recovery from hypogene samples is reasonable based on the testwork to date. Prediction of copper recovery for surface-linked-oxidation supergene samples applicable to Kansoko is more complex and variable. A separate method of copper recovery prediction for Kamoa supergene mineralisation uses measured ASCu assay values to reduce the recovery, where this is deemed necessary. It should be noted that the lack of surface supergene mineralisation at Kakula makes this issue irrelevant for that deposit.

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 Phase 6 tests. The IsaMill tests on a Kansoko sample has been used as an estimate of milling requirements for the Kakula PEA. However, the large proportions of difficult-to-grind mica minerals (muscovite and biotite) at Kakula mean that another signature plot test is required for Kakula, sometime before advancing the project to PFS.

An ongoing feature of the Kamoa-Kakula Project has been the changing target mine locations and mine priorities through the phases. The work performed to date is appropriate for the Kamoa 2017 PFS mine plan but will not be adequate should the mine plan change to incorporate significant mineralisation that has not yet been tested for either comminution or flotation response.

Compared to the Kamoa mineralised zones, the Kakula deposit has less variability in copper mineralisation, a low and consistent arsenic content and effectively equivalent comminution properties. The flowsheet design work performed for Kansoko has been proven as well suited to the Kakula samples and no major flowsheet changes (apart from those that are needed to accommodate high-grade feed) are currently envisaged for Kakula.

14 MINERAL RESOURCE ESTIMATES

14.1 Key Assumptions/ Basis of Estimate

The Kamoa and Kakula Mineral Resource models are two separate models within the Project area. The Kamoa Mineral Resource model was previously reported in the January 2018 Kamoa-Kakula 2017 Development Plan Technical Report. The Kakula Mineral Resource model described in this Report is an updated model incorporating significant additional drilling at Kakula.

The effective date of this updated Kamoa-Kakula Mineral Resource is 23 February 2018. The qualified persons for the Kamoa-Kakula Mineral Resource estimates are Dr. Harry Parker, RM SME, and Mr Gordon Seibel, RM SME, employees of Amec Foster Wheeler. The Kamoa and Kakula Mineral Resource estimations were constructed by Mr. George Gilchrist, Pr. Sci Nat, Ivanhoe's Mineral Resources Manager. The Kakula resource estimation methodology is very similar to the method used for the Kakula January 2017 PEA. Nested grade shells using 1%, 2% and 3% TCu thresholds were utilised to capture the considerably thicker, bottom-loaded mineralised intervals, and to honour the laterally continuous high-grade zones that present significant opportunities for mine planning. Higher-grade samples are well supported laterally and vertically, and hence no capping was applied. An inverse distance to the third power (ID3) interpolation method was used to reflect sharper changes between the higher-grade and lower-grade mineralisation. In addition to length weighting, SG weighting was used during the estimation of the 3% TCu grade shell to reflect the higher bulk densities of the higher-grade TCu mineralisation.

The following adjustments to the Kakula modelling approach have been made to account for changes identified at Kakula West:

- An anisotropic search aligned at 115° (south-eastern areas), 105° (central areas), and 065° (western areas) was used during estimation to honour the spatial anisotropy of the distributions of TCu grades and lithological thicknesses.
- Assaying for Acid Soluble Copper (ASCu) was discontinued early in the Kakula exploration programme due to the consistent low ASCu/TCu ratios. As a result, insufficient ASCu data exist across the full extent of the Kakula deposit to allow for reliable ASCu estimation.

Amec Foster Wheeler considers the Mineral Resource models and Mineral Resource estimates derived from those models to be consistent with industry best practices (CIM, 2003) and to conform to the requirements of CIM Definition Standards (2014).

14.2 Selective Mineralised Zones (SMZ)

14.2.1 Kamoa

The Mineral Resource estimate used 776 drillhole intercepts, which include drillholes within the mining lease, but excludes drillholes at Kakula and on the Kamoa and Makalu domes; these domes are areas where the favourable Ki1.1.1 stratigraphic unit is not present, or where the mineralisation has been completely leached. Included in the 776 drillholes are 16 twin holes (where the spacing between drillholes is <25 m) and six wedge holes. These drillholes were used in the estimation, and weightings assigned to these drillholes during estimation were scrutinised to ensure negative weights did not create estimation biases due to clustering that can result with close drillhole spacings.

Collar, survey, assay, stratigraphy and SG data were exported from the Ivanhoe acQuire database as a series of csv files, imported into Datamine Studio 3 mining software, and combined to form a desurveyed drillhole file. The entire Ki1.1.1 and Ki1.1.2 units were estimated, and if distinct mineralised zones within these units could be identified, they were modelled and estimated separately. The drillhole file was exported to Excel, and the SMZ selections in each hole were manually chosen.

In general, the SMZ selection was based on the 1.0% cut-off traditionally used in the 2D models. The basal contact of the SMZ is usually sharp and easily defined using the traditional 1% TCu cut-off. In areas with gradational grade profiles (typically the top contact), a lower cut-off approaching 0.5% TCu was used, as a 1% TCu cut-off would locally truncate the gradational grade profile. Since the grade profile is often a function of the localised development of siltstone or sandstone layers, these layers were evaluated during the SMZ coding. The nature of the grade profile and the characteristics of surrounding drillholes are also a key consideration to ensure that the defined top and bottom contacts of the SMZ in any specific drillhole matched the same part of the grade profile as the top and bottom contacts of the SMZ defined in surrounding holes. This was to give consideration to the transformation and estimation methodology used, where the transformation is specifically designed to match the grade profile from one drillhole to the next to ensure the appropriate vertical position of the samples is correct for selection in the kriging search neighbourhood.

Two main mineralised zones were identified, the Upper SMZ and Lower SMZ (Figure 14.4). These SMZs occupy distinct positions vertically, and lateral extents are largely controlled by growth faults especially evident at Kansoko Sud (Figure 14.4). The Upper SMZ occurs across the majority of the Kamoa area north-east of the growth faults, and was locally subdomained in the Kansoko Sud area (Upper SMZ 2), where a bimodal grade distribution develops in response to changes in stratigraphy in a narrow zone (500 m wide) along the trace of the growth faults. The Lower SMZ occurs to the south-west.

Figure 14.1 Plan View Showing Lateral Distribution of the Three SMZs

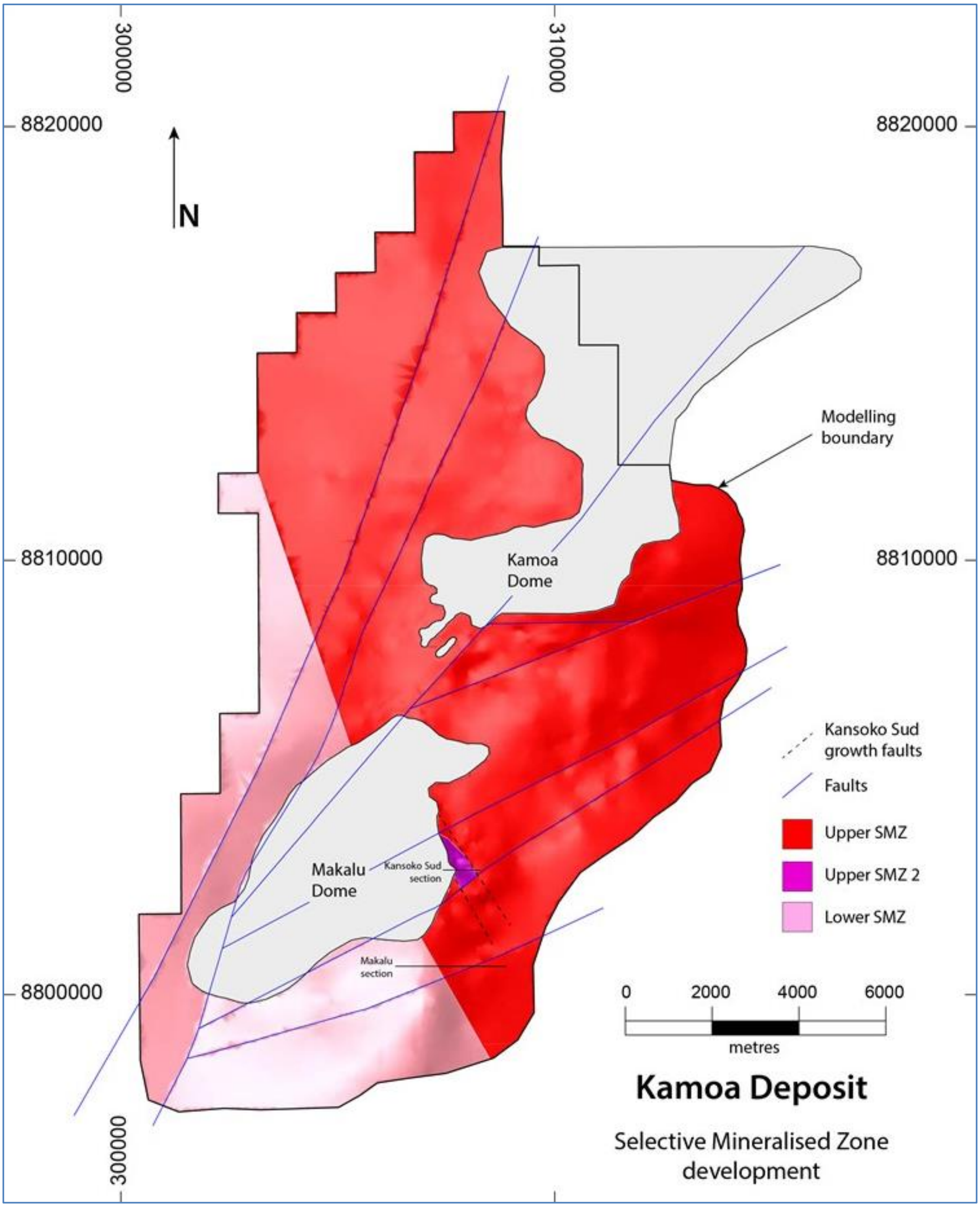


Figure provided by Ivanhoe, 2017.

The Upper SMZ is the most laterally continuous and best developed of the modelled minzones, and accounts for the vast majority of the Kamoa Mineral Resource. At Kansoko Sud, the intermediate siltstone (Ki1.1.1.2c) changes from a very narrow (or non-developed) siltstone to a much thicker zone of siltstone-sandstone-siltstone layers. The lower siltstone layer remains consistently mineralised (the Upper SMZ); the sandstone between the two siltstone layers is poorly mineralised, and the uppermost siltstone (Ki1.1.1.2.c) is moderately mineralised (Figure 14.2). To avoid smoothing of grades vertically during estimation, a separate SMZ was defined as a separate domain in these areas, incorporating grade in the sandstone and upper siltstone of the Ki1.1.1. 2c.

Figure 14.2 Relationship of the Two Upper SMZ Zones Developed at Kansoko Sud

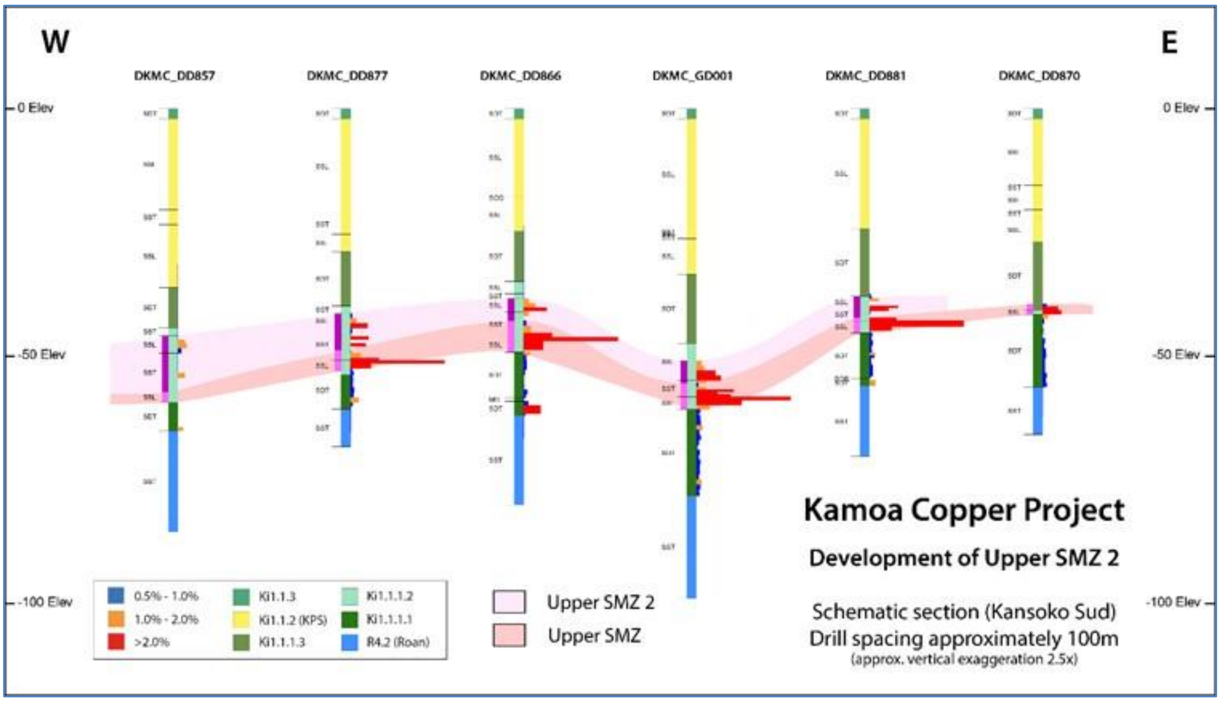


Figure provided by Ivanhoe, 2017. Section line (Kansoko Sud) shown in Figure 14.4.

South-west of the growth faults at Kansoko Sud, the mineralisation in the Upper SMZ weakens, and a separate mineralised zone develops at the base of the Ki1.1.1.1, close to, or on, the R4.2 contact. This Lower SMZ is generally lower-grade than the Upper SMZ, but is recognised in both the Makalu area and in Kamoa Ouest prospect area. A lack of drillholes in the southern portions of the Makalu prospect area make correlations with Kakula difficult; however, the mineralisation developed at Kakula occurs in the same stratigraphic position as the Lower SMZ. At Makalu, the lateral overlap between the Upper SMZ and Lower SMZ is approximately 800 m (Figure 14.3).

Figure 14.3 Relationship of the Upper SMZ and Lower SMZ Developed at Makalu

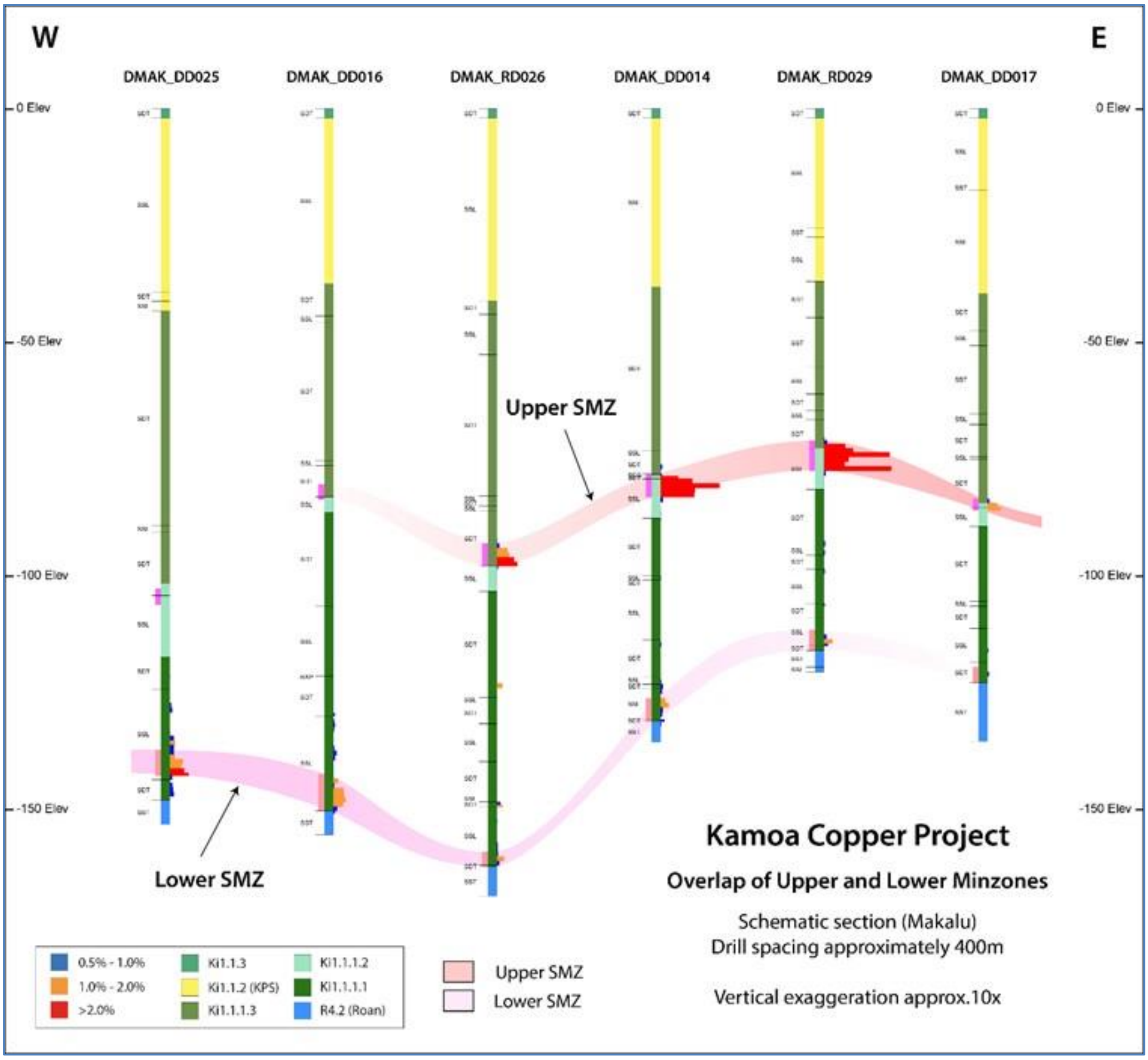


Figure provided by Ivanhoe, 2017. Section line labelled Makalu on Figure 14.1.

The assay file, with the SMZ selections flagged, was then imported into Datamine® Studio 3 mining software where it was combined with the collar and survey files. The SMZ selection fields were added to the de-surveyed drillhole files as a series of columns, with a value of '1' assigned where the samples were within a specific SMZ, and a default value of '0' for all other samples.

14.2.2 Kakula

At Kakula, a modified 2D approach was adopted whereby three TCu grade shells were coded to produce five SMZs using 1% (upper and lower, SMZ10U and SMZ10L respectively), 2% (upper and lower, SMZ20U and SMZ20L respectively), and central 3% TCu (SMZ30) cut-offs to represent the significantly thicker, bottom loaded, and more continuous higher-grade mineralisation (Figure 14.4).

Figure 14.4 Kakula: Typical Grade Profile for SMZ Definitions

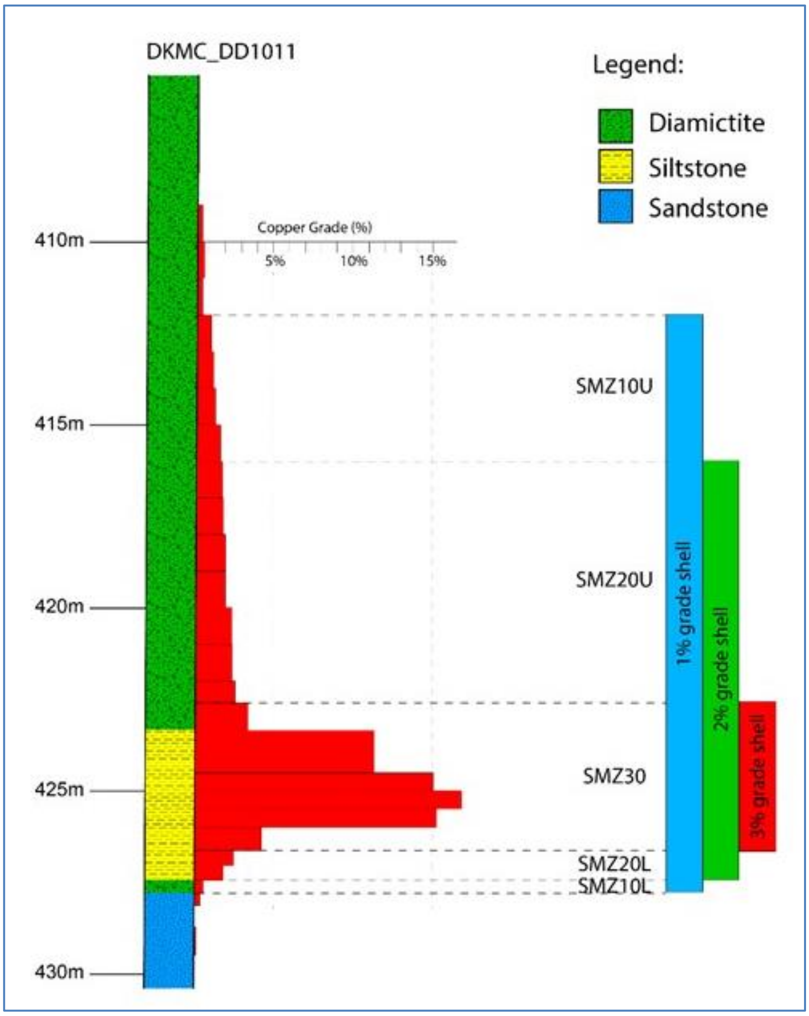


Figure provided by Ivanhoe, 2017.

The vertical continuity of the higher-grade mineralisation was evaluated by calculating and plotting the proportion of uncomposited samples within the 3% grade shell (SMZ30) that were above 3% TCu, Figure 14.5. Within the 3% TCu block model, 51% of the samples are greater than 3% TCu, showing that the vertical continuity in grade is exhibited on the individual-sample scale, and the high-grade composites are not a result of single high-grade intercept diluted with low-grade material.

Figure 14.5 Kakula: Proportion of Samples >3% TCu Within the 3% Grade Shell

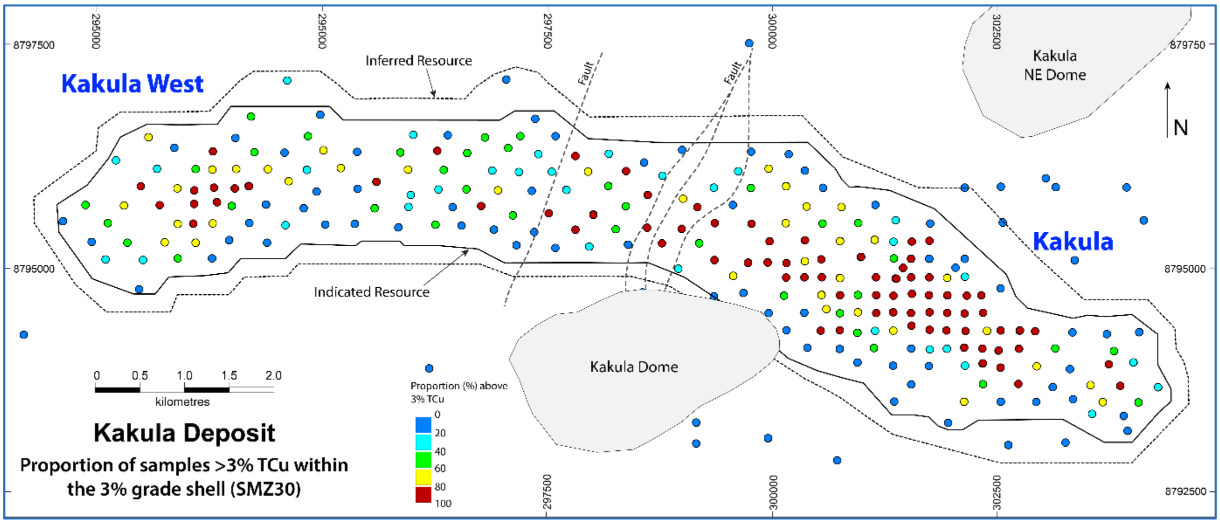


Figure provided by Ivanhoe, 2018.

An approximate minimum downhole length of 3 m was only applied to the 3% TCu grade shell (SMZ30) to reflect the minimum underground mining height. In the event that a drillhole did not meet the minimum grade ≥3% TCu and length greater than 3 m criteria, the highest grade 3 m composite was formed in the appropriate stratigraphic position. These lower-grade drillholes typically occur along the edge of the deposit, and were included to add lateral dilution.

If the 2% and 1% TCu grade shells above the 3% TCu grade shell (SMZ20U and SMZ10U respectively), and the 2% and 1% TCu grade shells below the 3% TCu grade shell (SMZ20L and SMZ10L respectively) were undefined, they were set to a default vertical thickness value of 0.001 m to ensure the grade shell thickness was correctly represented in the Mineral Resource model.

The average true thickness of the selective mineralised zone (SMZ) at a 1% TCu cut-off is 10.1 m in the Indicated Resources area and 6.7 m in the Inferred Resources area. At a higher 3% TCu cut-off, the average true thickness of the SMZ is 4.7 m in the Indicated Mineral Resources area and 3.3 m in the Inferred Mineral Resources area.

14.3 Domaining

14.3.1 Kamoa

Estimation domains at Kamoa were developed by combining the geological and mineralisation models using the stratigraphic and SMZ coding to create domains that honour both the vertical and lateral controls on mineralisation. Eleven domains are modelled (Table 14.1 and Figure 14.6). Contacts between domains are treated as hard contacts for resource estimation purposes.

Table 14.1 Domains Used for Grade Estimation with SMZ Domains Highlighted

Domain	Description
100	Ki1.1.2 (KPS) outside of any portions occurring within the Upper SMZ.
110	Portion of the Upper SMZ occurring within the KPS.
200	Where the Upper SMZ occurs in the Ki1.1.1, this is the zone between the top contact of the Upper SMZ and the base of the KPS (effectively represents unmineralised Ki1.1.1.3).
210	Portion of the Ki1.1.1.2c outside of the Upper SMZ.
220	Portion of the Ki1.1.1.2b outside of the Upper SMZ (very limited in its extent, in close proximity to growth fault/s at Kansoko Sud).
230	Portion of the Ki1.1.1.2a outside of the Upper SMZ (very limited in its extent, in close proximity to growth fault/s at Kansoko Sud).
300	The Upper SMZ where it occurs in the Ki1.1.1.
310	The second Upper SMZ to model the upper portion of the bimodal grade distribution in the Kansoko Sud area.
400	The zone between the bottom contact of the Upper SMZ and the top of the Lower SMZ or top of the Roan (effectively represents unmineralised Ki1.1.1.1).
500	The Lower SMZ.
600	A consistent 5 m thick portion of the uppermost part of the Roan (R4.2).

Figure 14.6 Schematic Illustrating the Vertical Position of the Estimation Domains (Localised Domain 220 and Domain 230 Excluded)

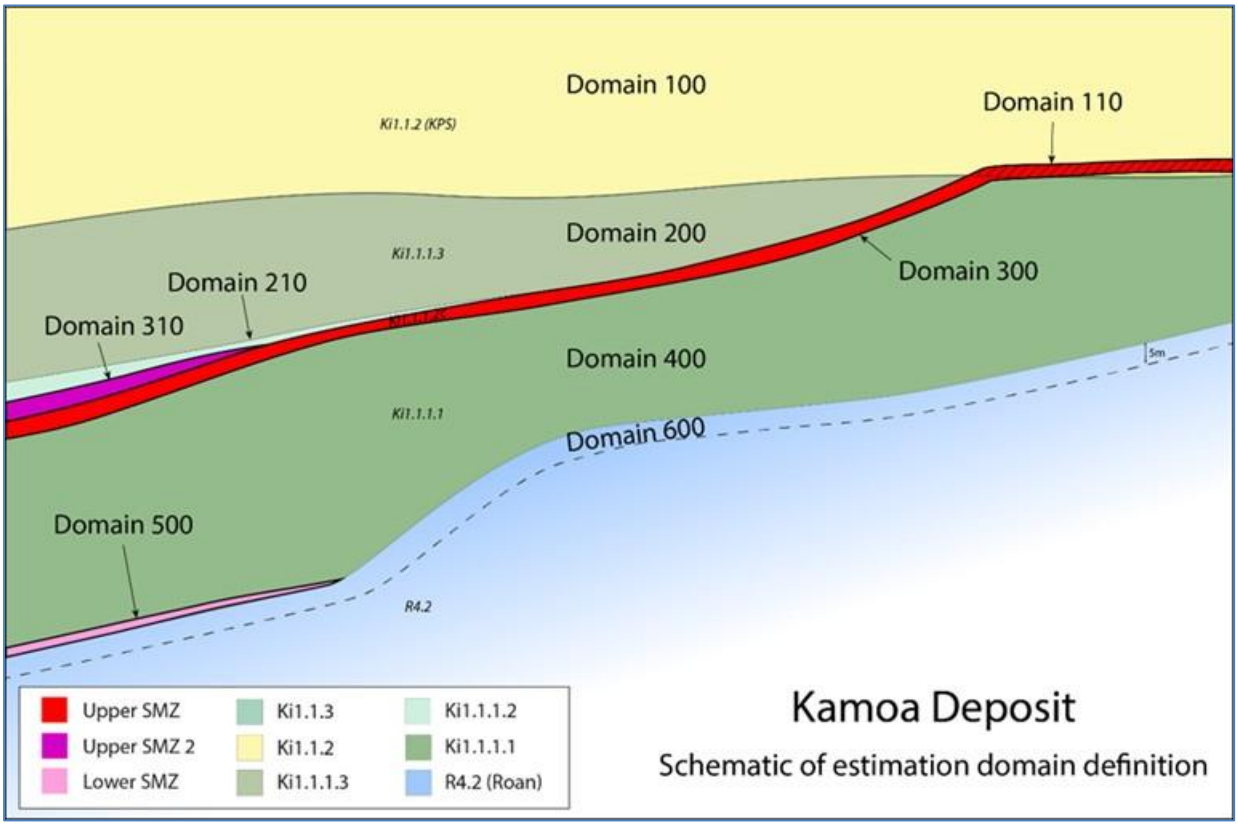


Figure provided by Ivanhoe, 2017.

14.3.2 Kakula

Given the 2D nature of the modelling, no vertical domains were applied. Three lateral domains have been defined in establishing the search parameters to allow elongated search ranges with an azimuth of 115° in the southeast portions of Kakula, an azimuth of 105° in the central portions, and an azimuth of 065° at Kakula West .

14.4 Top Capping

14.4.1 Kamoa

Samples are collected at a nominal 1 m sample length, with shorter intervals used to honour geological contacts. Outside of mineralised zones within the KPS, a 3 m composite sample has been submitted for analysis, primarily to determine the sulphur content within the KPS. The drillhole samples were first combined into 1 m composites, honouring domain contacts, and then capped.

Determination of the capping value was based on the following factors:

- Analysis of the histogram and log probability plots for each variable to determine where the grade distribution began to break apart.
- The coefficient of variation (CV), a measure of the standard deviation of the distribution divided by the mean, was assessed. CVs in excess of 1.3 were considered to represent strongly skewed distributions, or distributions with significant outliers that would require capping.
- Spatial analysis of higher-grades to determine if these grades were supported by other nearby high-grade samples both vertically and laterally.
- A lower capping threshold (as a proportion of the distribution) was applied to domains with limited data.
- Spatial review of the highest TCu grades showed the highest grades are typically clustered and show good connectivity between drillholes. As a result, they were not capped, or had a light capping applied.

Although the TCu CV for most mineralised domains is <1.0 (considered to be low), isolated high-grade samples may have too much local influence during estimation, and were capped. Top capping values were applied per domain prior to estimation. Table 14.2 shows the capping threshold, and the impact of top capping. Capping thresholds are also shown on the histograms in Figure 14.8. Top capping was not applied in Domain 310, as the domain is very constrained in its lateral extent, and high-grades are supported by neighbouring drillholes.

Table 14.2 Kamoa: Impact of Top Capping Per Domain on 1 m Composite Samples (SMZ10 Option)

Domain	Number of Samples	Capping Grade TCu (%)	Samples Capped	No Capping		With Capping	
				Mean (%)	CV	Mean (%)	CV
100	18,284	4.0%	6	0.05	3.97	0.05	3.60
110	441	10.0%	4	2.71	0.76	2.67	0.70
200	7,366	2.6%	7	0.14	1.85	0.14	1.77
210	1,356	2.5%	2	0.15	1.86	0.14	1.55
300	4,280	18.0%	6	2.56	0.91	2.56	0.90
400	8,969	2.0%	11	0.35	0.86	0.35	0.85
500	291	3.0%	3	0.93	0.78	0.91	0.72

14.4.2 Kakula

At Kakula top capping was studied using 1 m composites within the mineralised zone to assess if isolated high-grades samples existed, and whether these values should be capped to prevent over-estimation.

Kakula is characterised by its high-grade, and chalcocite-dominant mineralogy. Visual review of the higher-grade composites clearly showed that the higher-grade material holds together laterally on a 115° trend in the south-east portion of the deposit, along a 105° trend in the central portion of the deposit and along a 065° trend in the western portion of the deposit (Figure 14.7), and is constrained vertically by the basal siltstone. In addition, histograms and log probability plots for the 3% TCu grade shell show little breakdown in the grade distribution at higher-grades, and the distribution has a low CV value of approximately 0.6. TCu variograms have a low relative nugget effect (<10%) and long ranges (2,000 m or longer) along the 115°, 105° and 065° trends. Based on the strong support for the continuity of the higher-grades, and the modelling constraints used, no top capping was applied to samples used in the Kakula estimate.

Figure 14.7 Kakula: Visual Top Capping Analyses with TCu grades >8%, >10%, 12%, and >14%

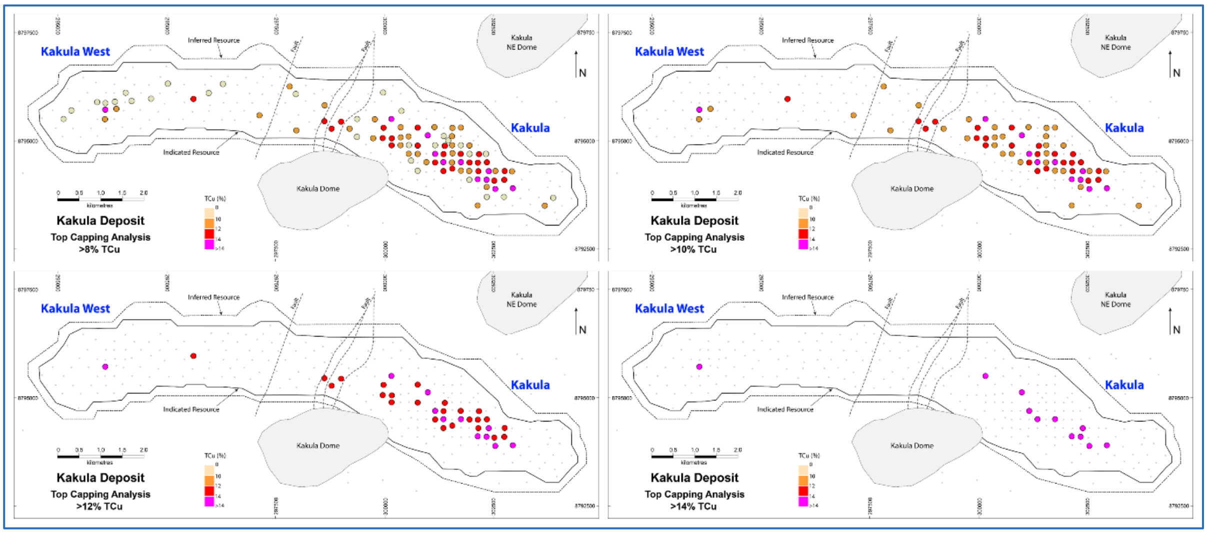


Figure provided by Ivanhoe, 2018.

14.5 Exploratory Data Analysis (EDA)

14.5.1 Kamoa

Composite statistics for 1 m samples for TCu and S are summarised in Table 14.3, and displayed graphically as histograms and log probability plots in Figure 14.8 and Figure 14.9. The distribution of TCu grades within the mineralised zones is positively skewed but generally well constrained with few outliers. Higher-grades are generally clustered and honour lithological or structural controls. The relationship between TCu grades and ASCu grades is shown in Figure 14.12.

SG values show very little variability, with distributions approximating normal distributions. Distributions per domain are slightly offset relative to one another depending on the dominant lithology of the domain (Figure 14.10). The KPS (Domain 100) is primarily shale, with an average SG of 2.79. Domains 200 to 500 are hosted within diamictite or intercalated siltstones, with average SG values of 2.60 to 2.67. The Upper SMZ (Domain 300) has a slightly higher SG of 2.73, likely due to the denser sulphide mineralisation. The porous R4.2 sandstone (Domain 600) has the lowest average SG of 2.51.

Sulphur grades are elevated in the Ki1.1.2 due to high concentrations of pyrite within the siltstone (Table 14.3). Sulphur grades are also elevated in the mineralised domains, most notably domains 110 and 310 where chalcopyrite dominates. A variety of sulphide species occur within Domain 300 (Figure 7.34), with bornite and chalcocite lowering the overall sulphur grade. Domain 500 is chalcocite-dominant, hence the lower sulphur grades. Overall, sulphur values are positively skewed (Figure 14.11). Arsenic values at Kamoa are very low, with approximately 65% of samples <0.001% As (Figure 14.11). Figure 14.12 shows a plot of ASCu versus TCu. In general, most samples have a ratio of ASCu/TCu of 10% or less (representative of sulphides where a small amount will dissolve in sulphuric acid), and very few have a ratio of over 30%, which would normally trigger selection of reagents that would coat the copper oxide minerals to make them float.

Table 14.3 Kamoa: 1 m Composite Statistics for Each Domain Option (Uncapped Data)

Variable	Domain	Number of samples	Minimum	Maximum	Mean	Standard Deviation	CV
TCu (%)	100	18,293	0.00	8.42	0.05	0.18	3.86
	110	**441**	**0.03**	**19.40**	**2.71**	**2.05**	**0.76**
	200	7,412	0.00	4.54	0.14	0.26	1.84
	210	1,373	0.00	4.16	0.15	0.23	1.60
	220	397	0.00	1.31	0.12	0.18	1.49
	230	123	0.04	1.66	0.37	0.26	0.72
	300	**4,246**	**0.01**	**22.72**	**2.58**	**2.33**	**0.90**
	310	**238**	**0.00**	**18.15**	**1.76**	**2.47**	**1.40**
	400	9,022	0.00	4.87	0.35	0.30	0.86
	500	**300**	**0.03**	**4.99**	**0.93**	**0.72**	**0.77**
	600	4,072	0.00	1.40	0.04	0.09	1.93
S (%)	100	15,803	0.003	26.09	7.20	5.84	0.81
	110	**433**	**0.003**	**17.52**	**3.89**	**3.04**	**0.78**
	200	6,050	0.003	14.68	0.68	1.00	1.47
	210	1,221	0.003	16.45	1.71	3.09	1.81
	220	396	0.003	7.92	0.56	0.98	1.77
	230	121	0.003	3.72	0.19	0.36	1.86
	300	**3,969**	**0.003**	**22.33**	**1.61**	**1.66**	**1.03**
	310	**237**	**0.010**	**16.61**	**3.28**	**3.72**	**1.14**
	400	8,497	0.003	1.70	0.11	0.12	1.07
	500	**237**	**0.003**	**1.18**	**0.16**	**0.17**	**1.06**
	600	3,457	0.033	3.10	0.05	0.12	2.59

Note: Mineralised domains are in bold.

Figure 14.8 Kamoa: Histograms of 1 m Composites for TCu (%) for All Mineralised Domains

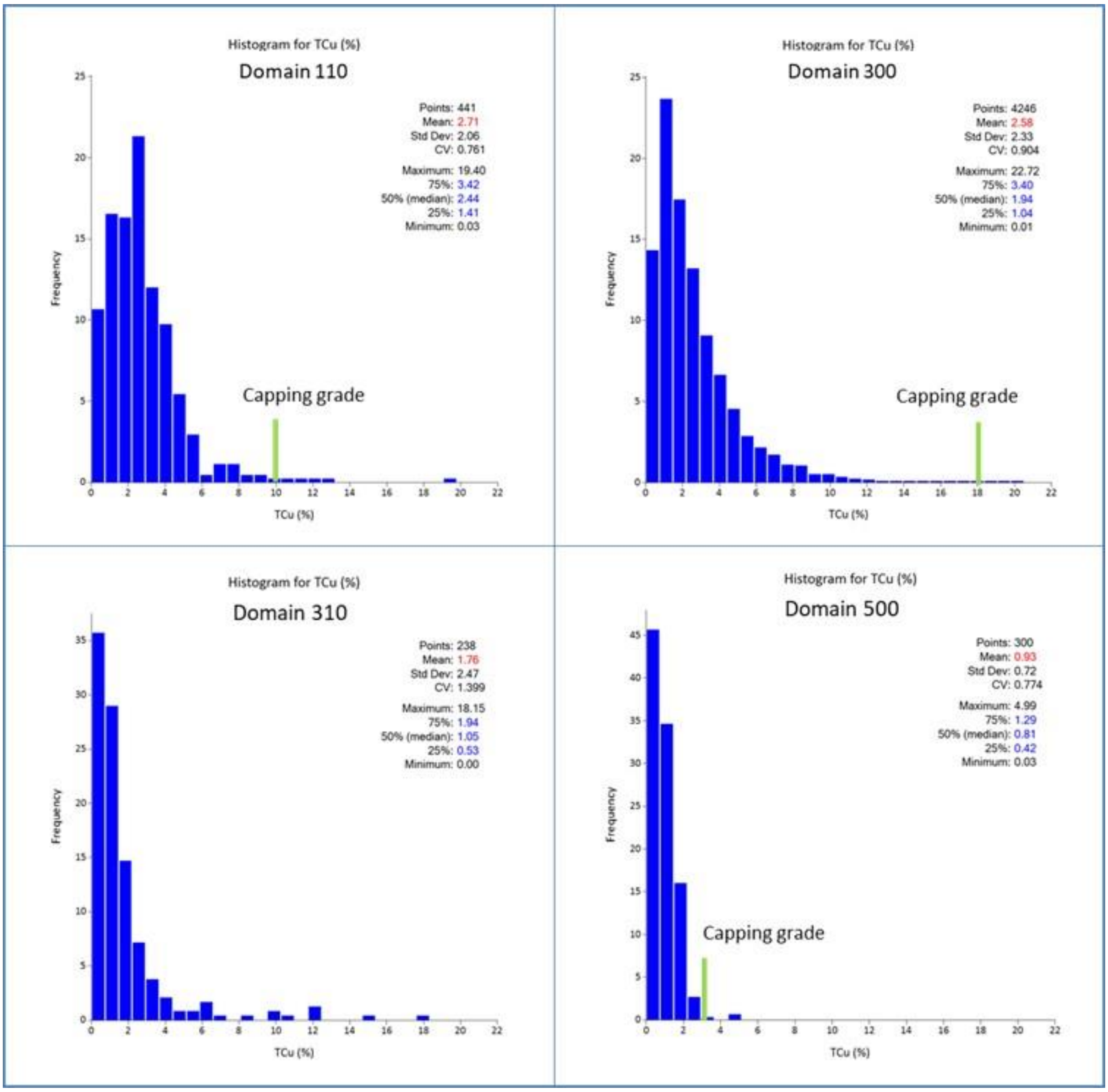


Figure provided by Ivanhoe, 2017. Green lines represent the top capping applied per domain, no capping was applied to Domain 310.

Figure 14.9 Kamoa: Log Probability Plots of 1 m Composites for TCu (%) for All Mineralised Domains

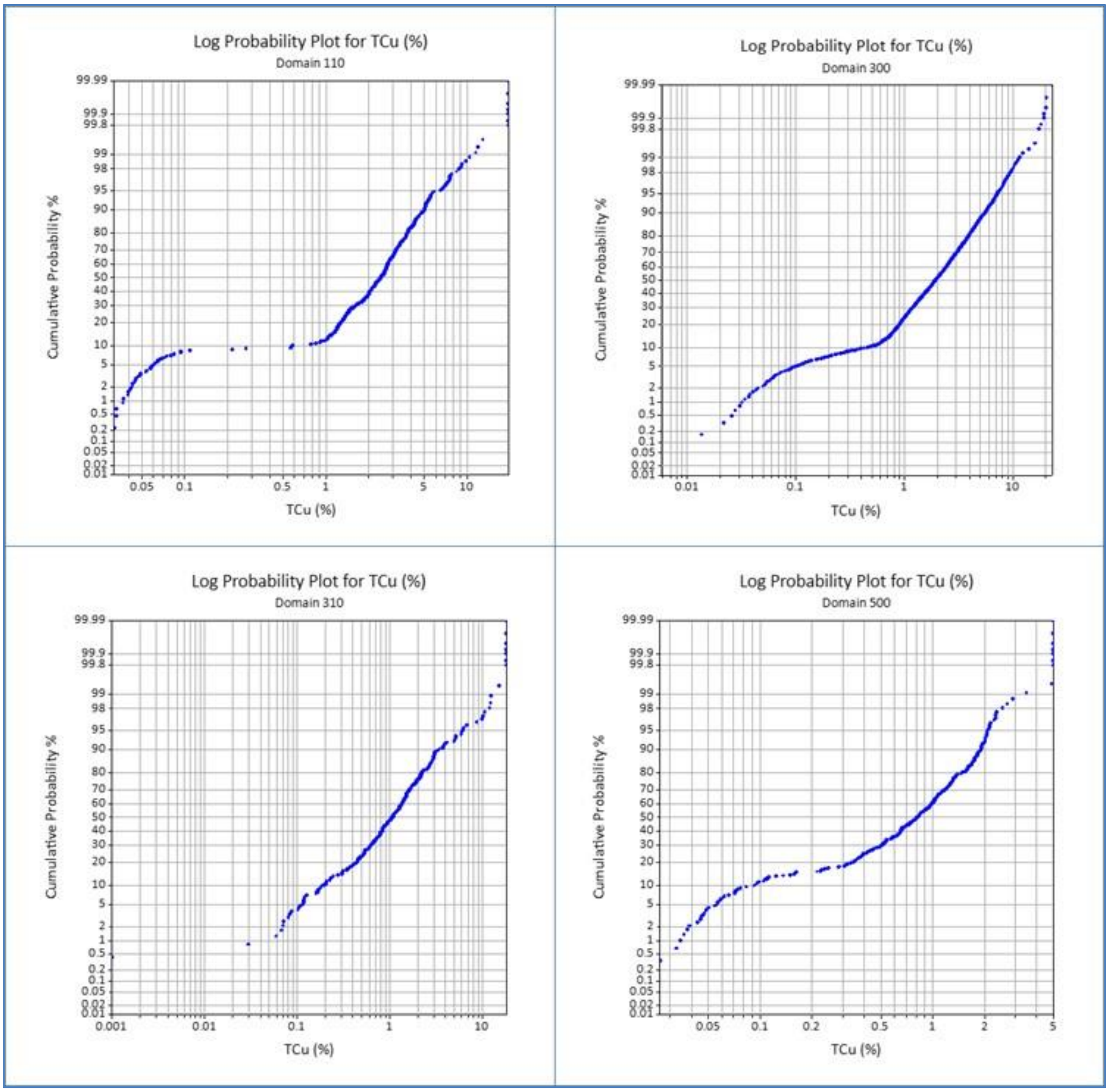


Figure provided by Ivanhoe, 2017.

Figure 14.10 Kamoa: Specific Gravity Values for a Selection of Lithologically Distinct Domains

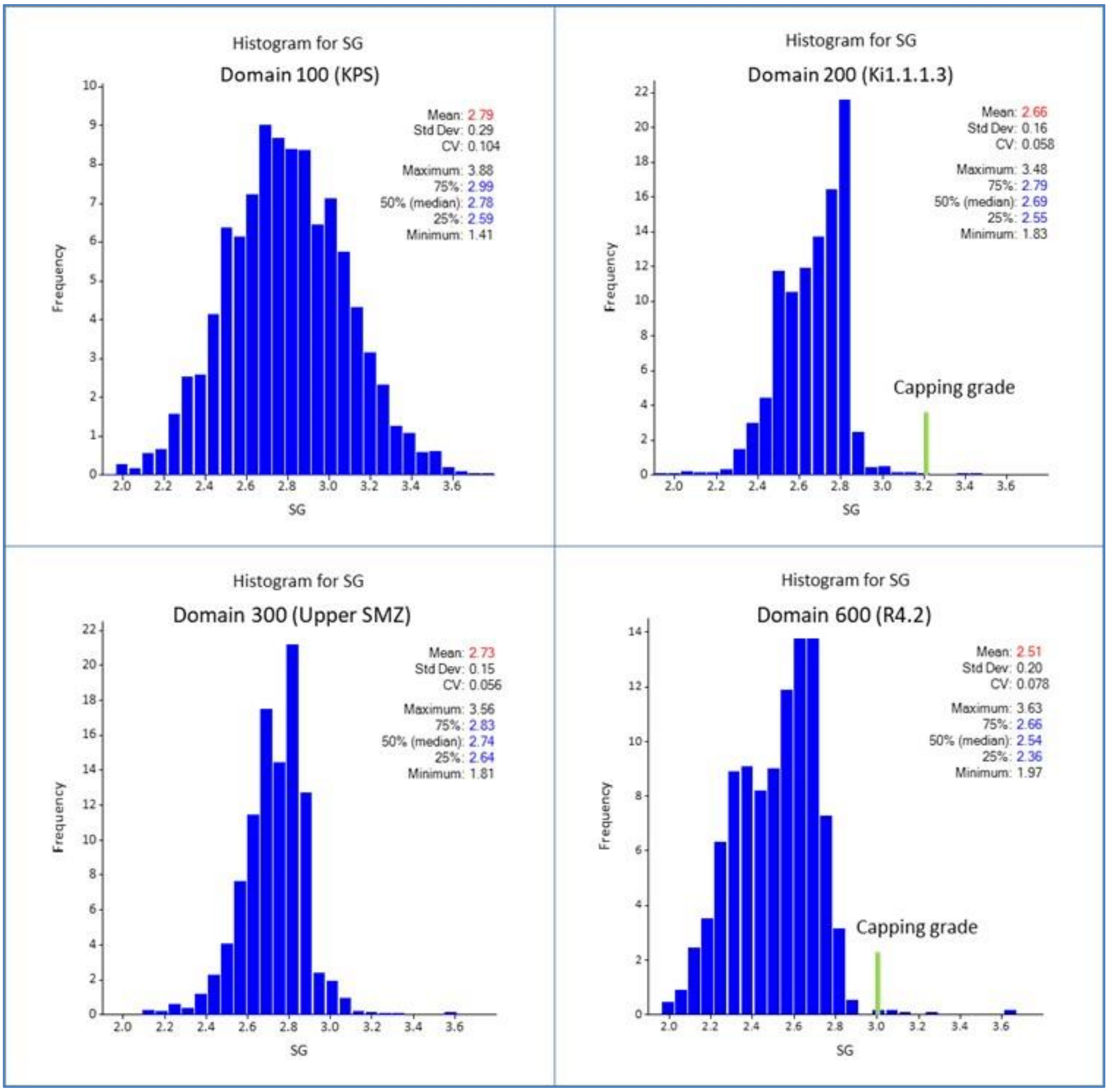


Figure provided by Ivanhoe, 2014; histogram and probability plot (weighted by true thickness).

Figure 14.11 Kamoa: Histograms and Log Probability Plots for Sulphur and Arsenic Values for Mineralised Domains (Domains 110, 300, 310 and 500)

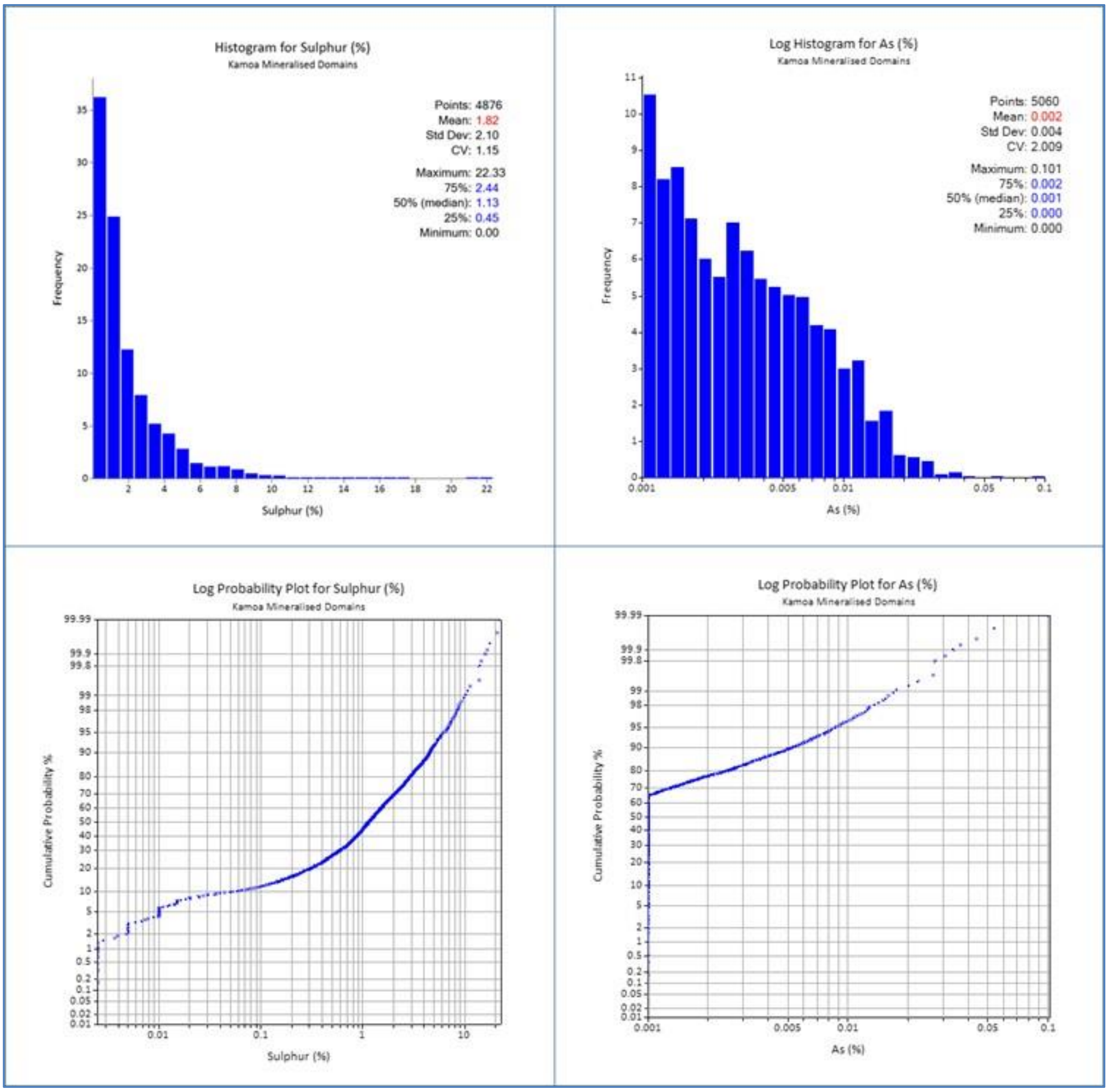
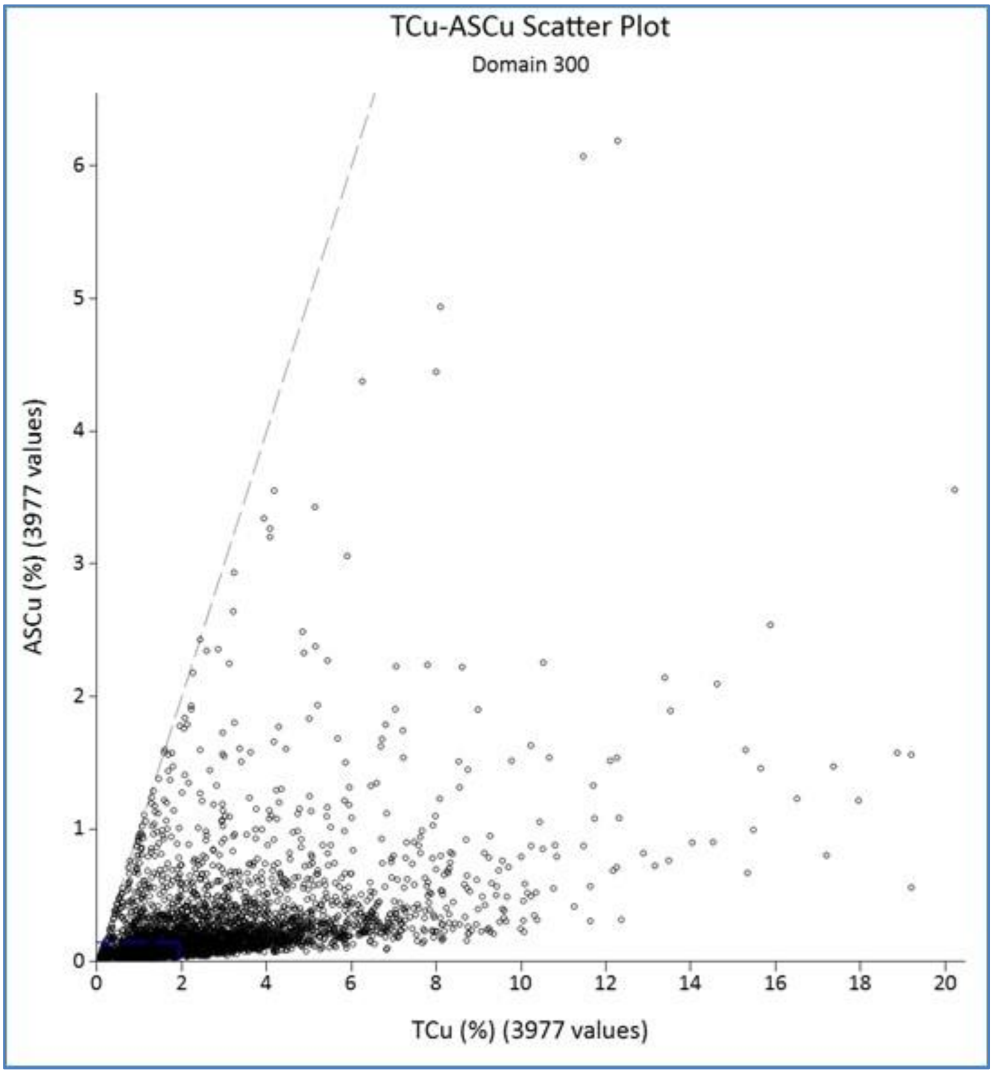


Figure prepared by Ivanhoe, 2017.

Figure 14.12 Kamoa: TCu:ASCu Values for Domain 300



14.5.2 Kakula

Exploratory data analysis was undertaken on 1 m composite samples and composites across the full width of each individual SMZ that was used in the grade estimation for each grade shell. A total of 136 drillholes were used in the modelling; however, only 121 of these occur within the Kakula Inferred Mineral Resource outline (also includes Indicated Mineral Resources). Five drillholes are located at Kakula West, three drillholes are located around the south-western edge of the Inferred outline, and seven drillholes are located north-east of the Inferred outline.

SMZ composite summary statistics for total copper (TCu), true thickness (TT) and total copper times true thickness (TCuxTT) for each of the five SMZs are presented in Table 14.4. Histograms and log probability plots for the 1 m composites for the key SMZs (SMZ10U, SMZ20U, and SMZ30) are shown in Figure 14.13, Figure 14.14, Figure 14.15, and plots for the true thicknesses of SMZs in the 3% TCu grade shell are shown in Figure 14.16. A scatter plot of true thickness versus TCu for the 3% TCu SMZ (where it was observed that at the higher true thicknesses, grade goes down as the true thickness increases) is shown in Figure 14.17. The same scatter plot for the 1% TCu upper grade shell (Figure 14.18) is shown to highlight that no obvious relationship exists between true thickness and grade within the narrow grade range of the 1% TCu upper grade shell.

Table 14.4 Composite Statistics for each SMZ at Kakula

Variable	SMZ	Number of Samples	Minimum	Maximum	Mean	Standard Deviation	CV
TCu (%)	SMZ10U	181	1.00	1.96	1.43	0.24	0.17
True Thickness (m)		271	0.00	25.62	3.14	4.47	1.42
TCu (%)	SMZ20U	129	1.96	2.92	2.31	0.20	0.09
True Thickness (m)		271	0.00	15.01	1.98	3.03	1.53
TCu (%)	SMZ30	271	0.08	12.21	4.17	2.72	0.65
True Thickness (m)		271	1.90	21.03	4.31	2.55	0.59
TCu x TT		271	0.22	110.68	21.05	21.79	1.04
TCu (%)	SMZ20L	68	2.00	2.96	2.40	0.26	0.11
True Thickness (m)		271	0.00	9.72	0.28	0.79	2.86
TCu (%)	SMZ10L	117	1.01	1.99	1.41	0.27	0.19
True Thickness (m)		271	0.00	23.46	0.67	1.74	2.58

Note: For the 1% (SMZ10U, SMZ10L) and 2% (SMZ20U, SMZ20L) grade shells, the TCu grade is left absent where the grade shell is not developed, but the thickness is set to 0.001 m, which is taken into account for the true thickness statistics.

Figure 14.13 Kakula: 1 m Composite TCu (%) for 1.0% TCu Upper Grade Shell (SMZ10U). Histogram and Probability Plot

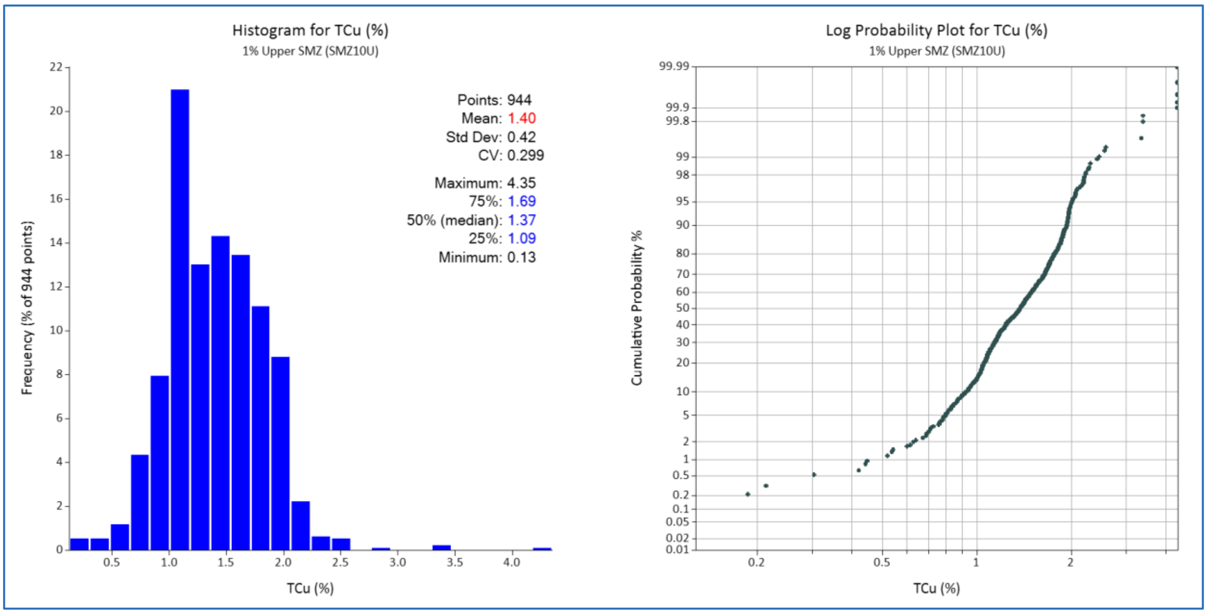


Figure provided by Ivanhoe, 2018.

Figure 14.14 Kakula: 1 m Composite TCu (%) for 2.0% TCu Upper Grade Shell (SMZ20U). Histogram and Probability Plot

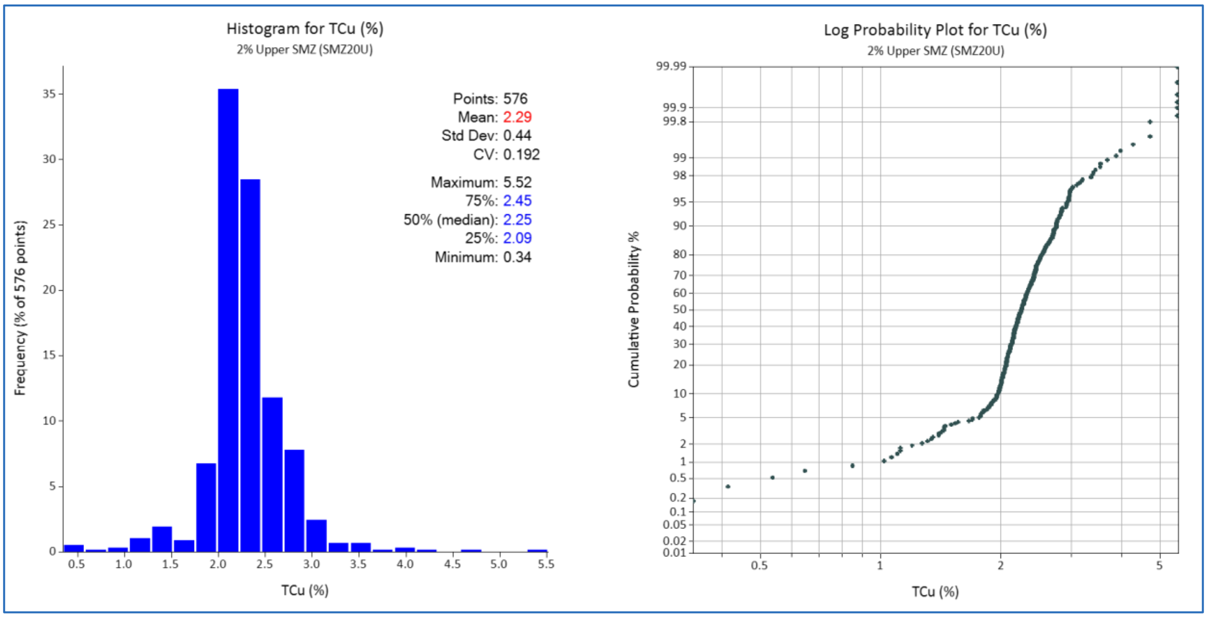


Figure provided by Ivanhoe, 2018.

Figure 14.15 Kakula: 1 m Composite TCu (%) for 3.0% TCu Grade Shell (SMZ30). Histogram and Probability Plot

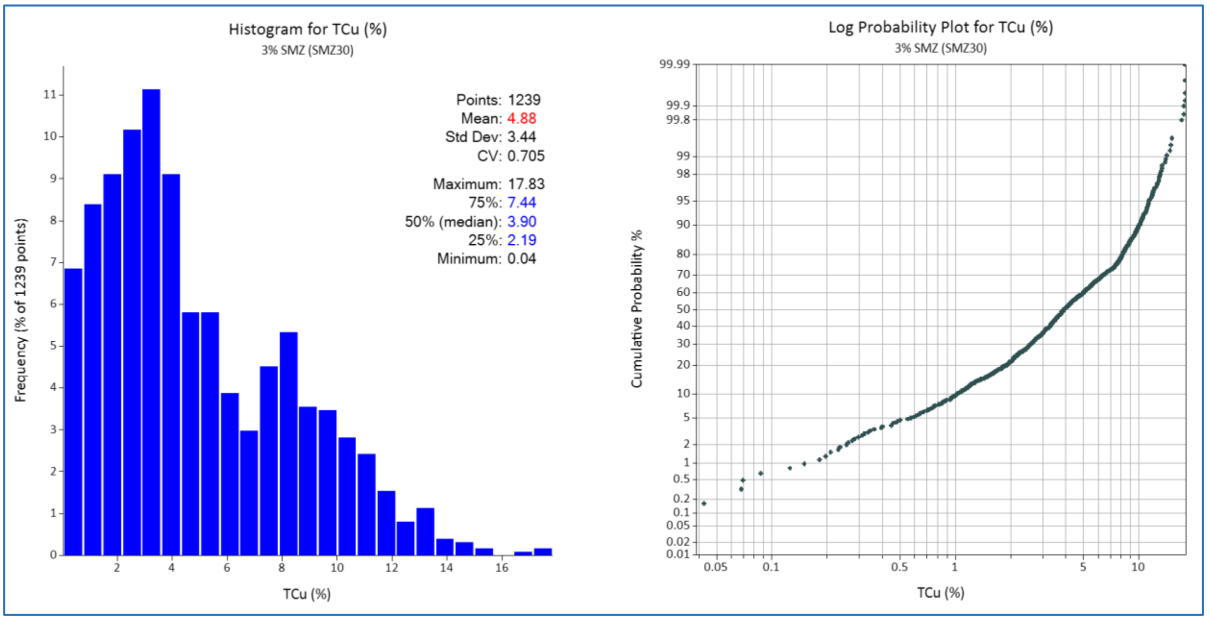


Figure provided by Ivanhoe, 2018.

Figure 14.16 Kakula: Full Width SMZ Composite True Thickness (m) for 3.0% TCu Grade Shell (SMZ30). Histogram and Probability Plot

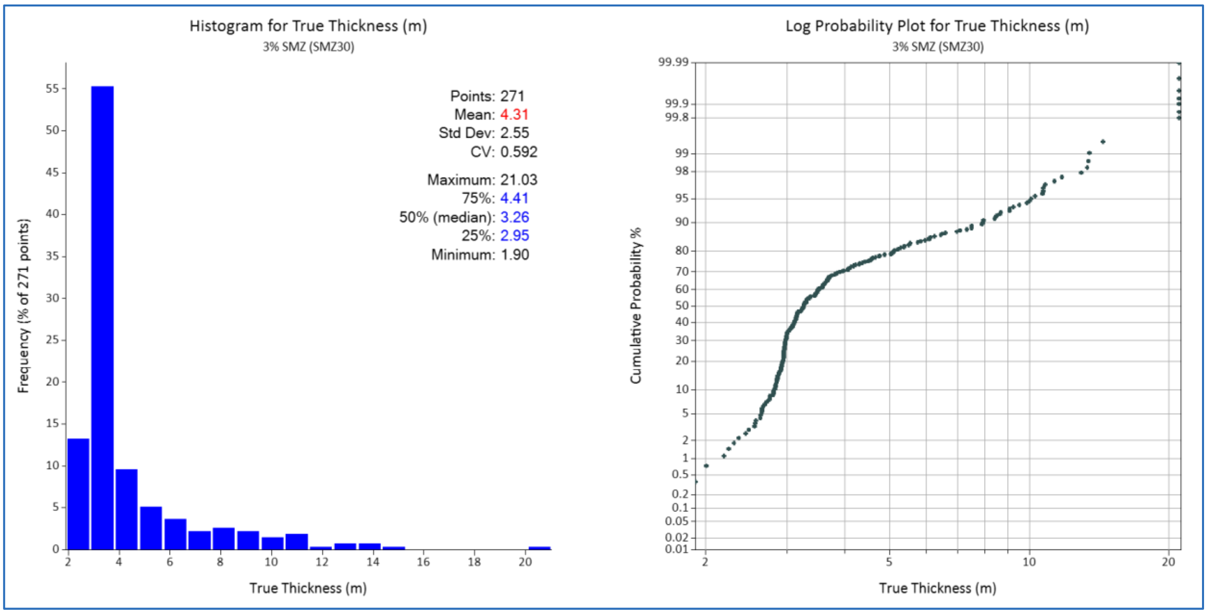


Figure provided by Ivanhoe, 2018.

Figure 14.17 Kakula: Scatter Plot of TCu (%) Versus True Thickness (m) for the 3% TCu Grade Shell

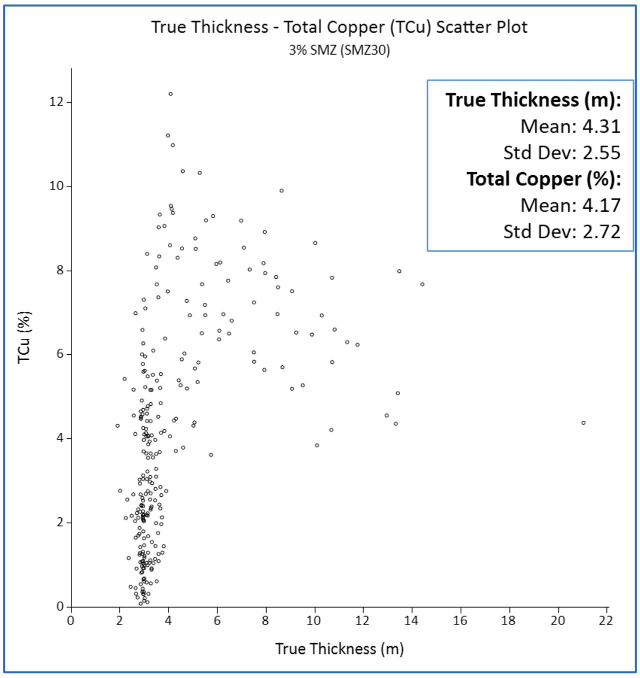


Figure Provided by Ivanhoe, 2018.

Figure 14.18 Kakula: Scatter Plot of TCu (%) Versus True Thickness (m) for the 1% Upper Grade Shell (SMZ10U)

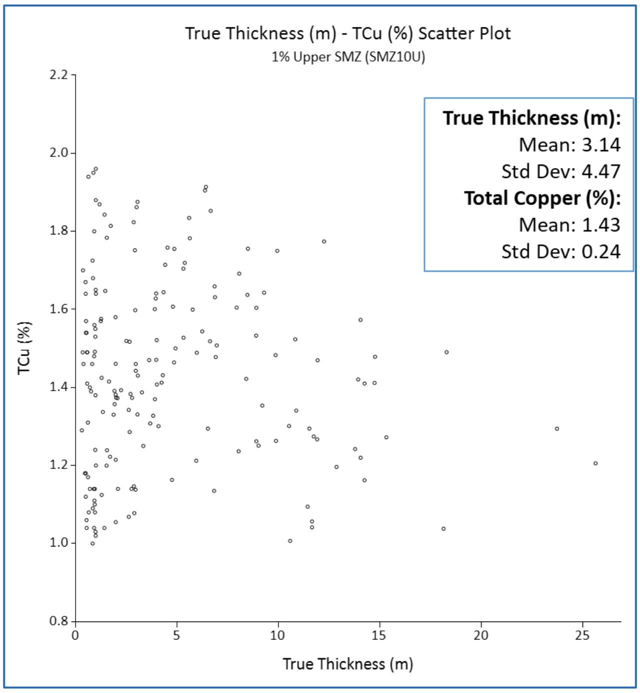


Figure provided by Ivanhoe, 2018.

Higher SG values in the higher-grade zones were recognised early on in the Kakula exploration programme, and SG measurements were collected on whole core for each sample interval that was assayed. Initial holes (prior to DKMC_DD1002) lack a full set of SG data. Since there is a strong relationship between TCu grade (%) and SG, a regression was performed (Figure 14.19) and used to assign an SG value to those samples with missing SG values.

Figure 14.19 Kakula Scatter Plot of Total Copper (%) and SG Values for the 3% TCu Grade Shell (SMZ30), with Outliers Removed

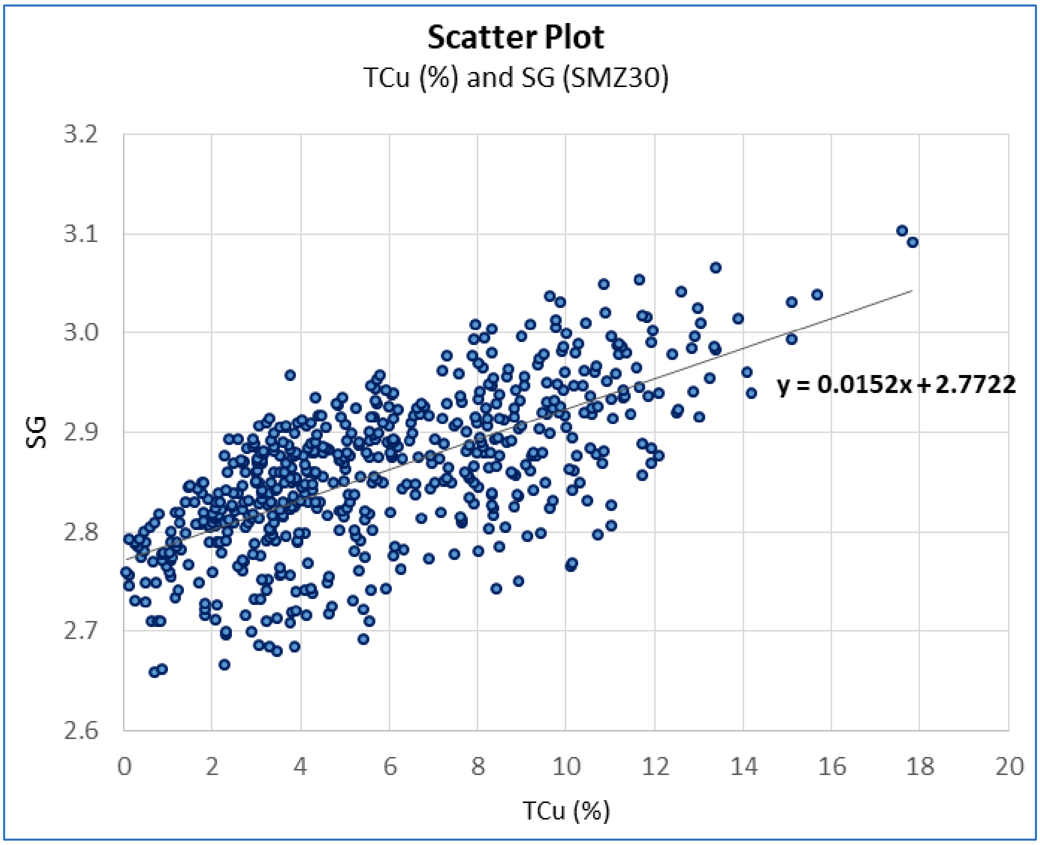


Figure provided by Ivanhoe, 2017.

14.6 Statistics Observations

14.6.1 Kamoa

Within the mineralised domains, TCu grades (refer to Figure 14.8) are positively skewed, but well constrained.

No clear relationship is evident between TCu and acid-soluble copper (ASCu). Higher ASCu grades are usually highly localised and concentrated in only one or two holes, indicating an inability to distinguish sulphide and oxide mineralisation into separate domains. Geological and metallurgical studies of the sulphide species indicate that the bulk of the mineralisation at Kamoa is sulphide, with localised oxide mineralisation closer to surface and along dome edges.

SG values for individual domains approximate a normal distribution, or are very weakly negatively skewed, with the mean SG value per domain strongly influenced by the dominant host lithology. The CV for individual domains is low, typically 0.1 or lower.

14.6.2 Kakula

By nature of the SMZ definition, TCu grades are well constrained within the 1% and 2% grade shells, with very little variability. The bimodality of the 1 m composite samples within the 3% grade shell has been shown to relate to the lithological control (refer to Figure 7.40). Bimodality related to lateral variations in grade is typically resolved through defining separate estimation domains for individual grade populations. For Kakula, the bimodality of the 1 m composites is a function of the vertical grade profile, and once the 1 m composites have been combined into the full width SMZs, the bimodality is no longer evident.

Consideration should be given to creating a vertical domain boundary at the lithological contact between the basal siltstone and diamictite when developingthe proposed 3D model for the next resource estimate. A relationship between TCu and TT is evident for the 3% grade shell, with the thicker intervals containing lower-grades. A relationship between TCu and SG is also evident, whereby higher-grade intervals have a higher SG. For estimation purposes, this relationship is accommodated in the estimation by assigning a TT and SG weighting during estimation for the 3% TCu grade shell.

14.7 Structural Model

Although the 3D Kamoa Mineral Resource estimations were performed in transformed (dilated) space, the model was back-transformed into real space for mine planning.

Eight structures were defined at Kamoa using geophysical data, and lithological discontinuities interpreted from the drillhole data (refer to Section 7.3.4). These structures were then used to divide the model into nine structural zones with internally similar strikes and dips. For grade estimations, the blocks and drillholes were transformed to two dimensions representing each structural zone, with the SMZs allowed to be included across the structural domain boundaries in the estimation. The same approach was adopted at Kakula, with five structural blocks defined.

Amec Foster Wheeler concurs that this approach appears reasonable, as the faulting in most of the Project area appears to have occurred after the deposition of the mineralisation.

At this time, it is difficult to establish the dips of the interpreted faults, and/or to determine if they are a single fault plane or represent a fault zone. For the Kamoa resource model, the simplest interpretation of the faults was used, which assumed that the faults are single vertical planes. Fault intervals identified in drill core at Kakula have allowed a steep dip (approximately 75°) to be modelled for these faults. Other faults and/or fractured zones have been mapped, based on geophysics and observed broken core; however, the available data are too wide-spaced to establish the dip and extent of these faults. Structural information from the exploration drift recently completed at Kamoa should be evaluated and included in future resource estimates as it comes available. This will be a key piece of information in understanding the geometry of the mineralisation and its implication on the efficacy of the proposed mining methods and Project economics.

14.8 Surface and Block Modelling

14.8.1 Kamoa

Surface modelling and block model estimation were limited within perimeters defining the mineralised portions and permit boundaries of the Project. Two prominent domes, the Kamoa dome to the north and the Makalu dome to the south, were excluded from the modelling as they represent leached areas, or barren areas where the Roan sandstone (R4.2) crops out at surface.

The Mineral Resource area was subdivided into nine structural domains based on the structural model and coded with grade domains using wireframes that define the stratigraphic units and mineralised zones. A 3D block model was established using the parameters provided in Table 14.5.

Table 14.5 Kamoa: Block Model Parameters

Axis	Origin	Maximum	Block Size (m)	# Blocks
Easting (X)	300,000	315,000	50	300
Northing (Y)	8,797,000	8,821,000	50	480
Elevation (Z)	-300	1,600	1	1,900

To improve stationarity for grade estimation, both the drillholes and the block model were transformed to ensure that the vertical TCu grade profiles matched between drillholes. Typically, these profiles are bottom-loaded, with the higher-grades occurring at the bottom of the profile and grading upwards to lower and lower-grades towards the top of the profile. The transformation was performed by adjusting the Z-coordinate of the data to 'dilate' drillhole composites and blocks to the maximum vertical thickness of the SMZ for each domain. This ensures that the lower, middle and upper portions of the grade profile correctly align between drillholes (Figure 14.20).

Hard boundaries were used for individual stratigraphic and mineralisation domains (whereby only data within the domain are used), and soft boundaries were used for structural domains. Variography and estimation were completed in transformed space. The block models were transformed back to their original vertical location by setting the centroid of each block back to its original Z-coordinate.

Figure 14.20 Kamoa: Vertical Section Showing Untransformed Composites and Blocks (Top) and Transformed Composites and Blocks (Lower) for Domain 300, 3x Vertical Exaggeration

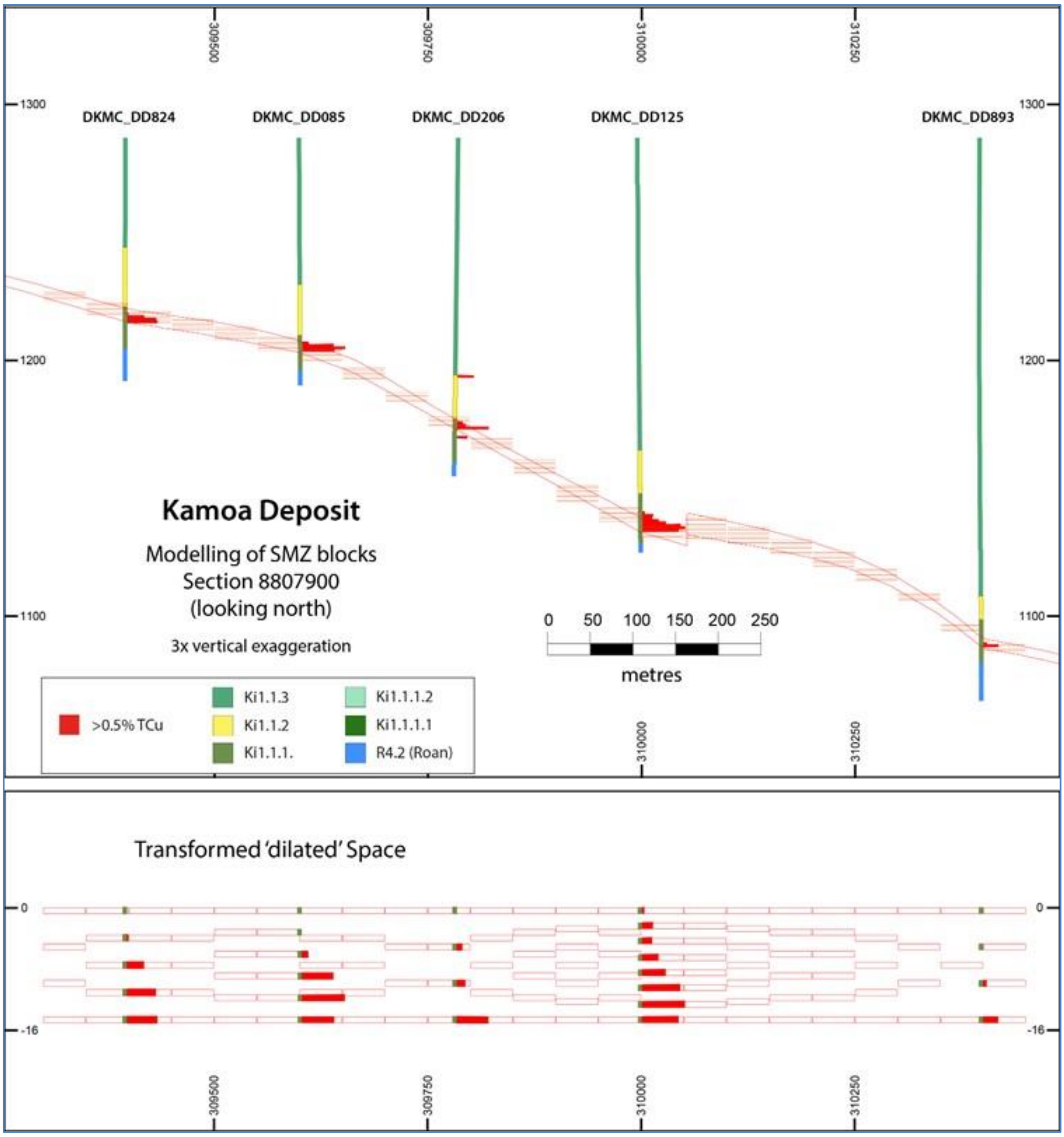


Figure prepared by Ivanhoe, 2017; Copper grade intensity shown by bars on right side of hole.

Transformed 1 m composites were used for variography. The variogram parameters were first optimised by performing sensitivity studies on the lag, angular tolerance, bandwidth and transform applied (normal-score transform) prior to modelling of the variogram. The vertical bandwidth was a key element given the vertical TCu trends evident in drillholes, and was typically set to a narrow interval. Downhole variograms of the transformed 1 m composites were used to determine the nugget effect (C0). The transformation dilates downhole samples, moving them further away from each other and potentially overstating continuity at short ranges. As a validation, downhole variograms of untransformed 1 m composite samples were also investigated and were found to be comparable. Variogram parameters for TCu for mineralised zones are summarised in Table 14.6, and variogram parameters for S for mineralised zones are summarised in Table 14.7.

Weak anisotropy at 115° is evident in Domain 110 (the mineralised portion of the KPS). A more robust, and stronger developed anisotropy at 145° is evident in the Upper SMZ (Domain 300) for both TCu (Figure 14.21) and S. This orientation matches that of the thickness changes evident in the stratigraphic units, and parallels the interpreted orientation of growth faults at Kamoa.

Table 14.6 Kamoa: TCu Variogram Parameters Categorised by Mineral Domain

Domain	Major Direction	Nugget C_0	C_1	Range (m)	C_2	Range (m)	C_3	Range (m)
110	115°	0.09	0.51	545	0.4	2025		
				400		1000		
				3		6		
300	145°	0.05	0.36	300	0.25	530	0.35	2300
				200		300		1100
				13		16		22
310	145°	0.16	0.4	172	0.44	1035		
				100		300		
				3		18		
500	145°	0.11	0.6	600	0.29	1600		
				500		1300		
				69		70		

Note: Domain 500 used variogram parameters from Domain 400.

Table 14.7 Kamoa: S Variogram Parameters Categorised by Mineral Domain

Domain	Major Direction	Nugget C_0	C_1	Range (m)	C_2	Range (m)	C_3	Range (m)
110	120°	0.06	0.49	395	0.46	1900		
				395		1300		
				6		7		
300	145°	0.04	0.47	640	0.49	1600		
				280		1100		
				29		37		
310	135°	0.10	0.66	210	0.24	950		
				200		340		
				10		12		
500	140°	0.11	0.43	320	0.22	1000	0.24	4500
				320		1000		1840
				89		119		143

Note: Domain 500 used variogram parameters from Domain 400.

Figure 14.21 Kamoa: Normal Score Major and Semi-Major Direction Variograms for TCu (Domain 300)

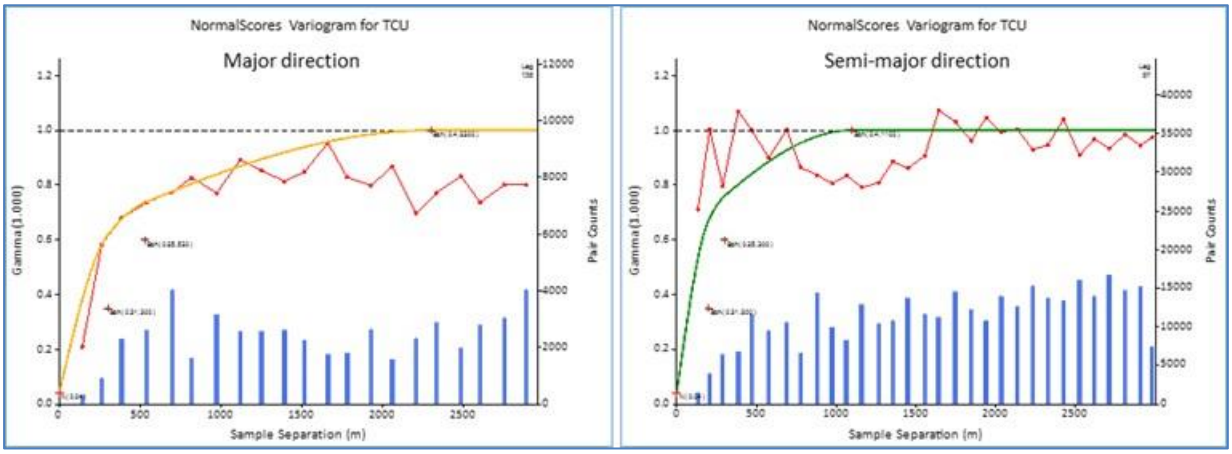


Figure prepared by Ivanhoe, 2017.

All grade variables (TCu, ASCu, As, Fe, and S) were estimated into each block using ordinary kriging (OK) interpolation for reporting, and an inverse distance to the second power (ID2) and nearest neighbor (NN) estimate were used for validation purposes. Estimation parameters are summarised in Table 14.8. Search parameters were adjusted for each variable within each domain based on the grade continuity evident from the variography. For all variables, if the block remained unestimated following the first search, the search was doubled in size. If necessary, this was again expanded by a factor of 2.5 for a third search.

Table 14.8 Kamoa: Estimation Parameters for TCu for all Mineralised Domains

Domain	Orientation			Search Range	Number of Samples Search Pass 1		Number of Samples Search Pass 2	
	Axis	Azimuth	Dip		Minimum	Maximum	Minimum	Maximum
110	X	115°	0°	1,000	4	12	4	8
	Y	0°	0°	500	4	12	4	8
	Z	0°	90°	5	4	12	4	8
300	X	145°	0°	1,100	4	12	4	8
	Y	0°	0°	600	4	12	4	8
	Z	0°	90°	5	4	12	4	8
310	X	145°	0°	450	4	12	4	8
	Y	0°	0°	200	4	12	4	8
	Z	0°	90°	5	4	12	4	8
500	X	145°	0°	1,200	4	12	4	8
	Y	0°	0°	1,000	4	12	4	8
	Z	0°	90°	5	4	12	4	8

A limit of a maximum of three samples from a single drillhole was used to ensure that at least two drillholes were used for any estimate, preventing any possible string effect occurring, where weights are preferentially assigned to the outermost samples when all samples used in an estimate are aligned in a row.

ASCu values are not available for every sample that contains a TCu value. This is particularly relevant in the Ki1.1.2, where only 21% of TCu samples have a corresponding ASCu value. Within the Upper SMZ (Domain 300), 94% of TCu samples have a corresponding ASCu value. To overcome this, an ordinary kriging estimation of TCu and ASCu using the search and variogram parameters for TCu was completed using only samples that contained both a TCu and ASCu value. Using this estimate, the ratio of TCu:ASCu was calculated. The final ASCu grade was then back-calculated from the TCu estimate (using all available TCu samples) and the calculated ratio.

14.8.2 Kakula

Surface elevation modelling and block model creation were limited by perimeters defining the unoxidised mineralised portions of the project. The Kakula North-East dome to the north and the Kakula dome to the south were excluded from the modelling as they represent leached barren areas. The extents of the Kakula model were defined by a broad rectangle constrained by the available drilling.

The Mineral Resource area was subdivided into five structural domains using the Kakula structural model. A digital terrain model (DTM) was first constructed through the centroid of the 3% grade shell (SMZ30) to define the geometry of the SMZ30 surface within each fault block domain. The elevations of 1% and 2% models above and below this surface were adjusted based on their estimated vertical thickness and their vertical distances above or below the SMZ30 model.

The DTM surface was constructed in Leapfrog Geo software using structural discs created using Datamine software that represent the local dip and dip direction of the SMZ30 surface around the centroid of each SMZ30 composite. To minimize DTM artefacts adjacent to the composites, the dip and dip directions of the structural discs were established by constructing an initial DTM of the Ki1.1.1-R4.2 contact in Leapfrog, importing this DTM into Datamine, and then estimating the dip and dip direction of the DTM surface into the centroid of each SMZ30 composite. The structural discs were then imported into Leapfrog Geo where they were used to control the local dip and dip direction of SMZ30 DTM near the centroids of the SMZ30 composites, while honouring the location of the centroids of the SMZ30 composites. Control points were also added along the edges of fault blocks and domes in areas of sparse data to ensure relatively consistent offset along the faults. The elevation of the SMZ30 block centroid was then set to the elevation of the SMZ30 DTM at the corresponding easting and northing co-ordinates.

A gridded seam prototype model was first established using 50 m x 50 m blocks in easting and northing, with a single block in elevation per SMZ using the parameters provided in Table 14.9.

Table 14.9 Kakula Block Model Parameters

Axis	Origin	Maximum	Block Size (m)	# Blocks
Easting (X)	291,000	306,000	50	300
Northing (Y)	8,791,000	8,798,000	50	140
Elevation (Z)	N/A	N/A	variable	1

The elevations of model blocks and composites were first flattened or set to an elevation of zero for resource estimation. In the 3% grade shell estimations (SMZ30), TCu and S were weighted by the TT and SG to reflect the higher densities of the higher-grade TCu mineralisation. For example, SG, TCu, and TT were multiplied together to obtain the variable SGxTCUxTT, and SG and TT were multiplied to obtain the weighting factor SGxTT. SGxTCUxTT, and SGxTT were then estimated into each block using inverse distance to the third power (ID3) using the estimation parameters summarised in Table 14.10. TCu was then calculated dividing the estimated SGxTCUxTT by the estimated SGxTT. As and Fe were weighted by true thickness only, as higher As and Fe values are not correlated with higher densities. No weightings were applied to any of the estimations in the 1% and 2% grade shells. There is no meaningful correlation between density, thickness and grade in these shells.

An ID3 interpolation method was selected to reflect sharper changes between the higher-grade and lower-grade mineralisation. An anisotropic search aligned at 115° (south-eastern areas), at 105° (central areas), and at 065° (western areas) was applied to honour the spatial anisotropy of the distributions of TCu grades and lithological thicknesses (Figure 14.23). The elevation of the SMZ10U, SMZ20U, SMZ20L, and SMZ10L were established after the grade estimations by stacking the models together using the SMZ30 model as a base and the vertical thickness of each grade shell model to form the complete Kakula model.

Table 14.10 Kakula: Estimation Parameters Used for the First Search

Search Domain	Orientation			Search Range	Number of Samples		Estimation Method
	Axis	Azimuth	Dip		Minimum	Maximum	
South-East	X	115°	0	1,000	4	18	ID3
	Y	25°	0	400	4	18	ID3
	Z	0°	-90	400	4	18	ID3
Central	X	105°	0	800	4	18	ID3
	Y	15°	0	500	4	18	ID3
	Z	0°	-90	500	4	18	ID3
Western	X	065°	0	800	4	18	ID3
	Y	155°	0	600	4	18	ID3
	Z	0°	-90	600	4	18	ID3

Note: An octant search was used, with samples from a minimum of two octants required for a block to be estimated.

Figure 14.22 Kakula: Varying Continuity Directions within the 3% Grade Shell (SMZ30) Defining Three Search Domains

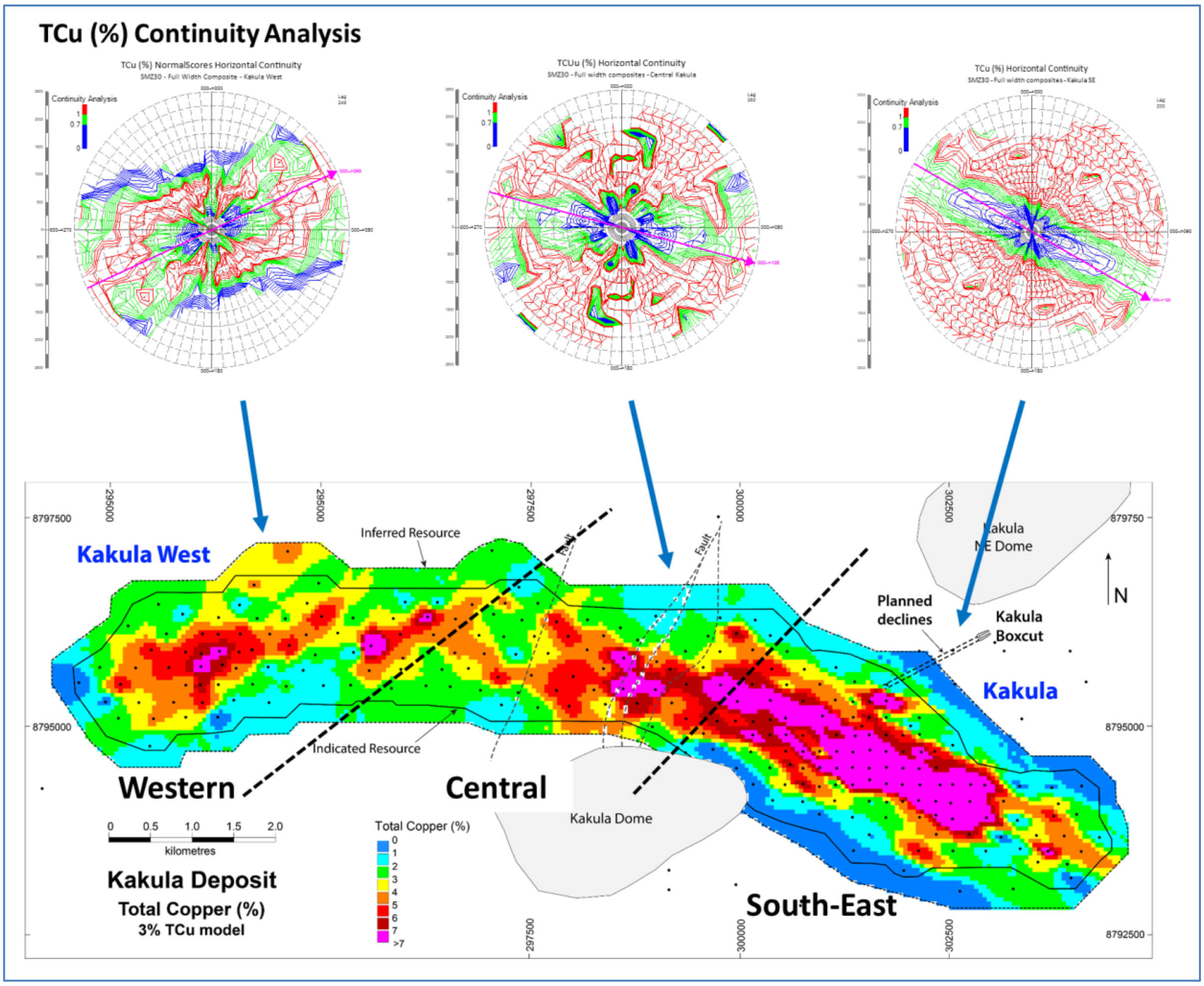


Figure prepared by Ivanhoe 2018.

14.9 Specific Gravity

For Kamoa, SG was estimated in transformed space using ID2, using only those SG samples that occurred within individual domain wireframes (refer to discussions in Section 11.5 on SG determinations). Search parameters were the same as those used for sulphur, with the minimum number of samples and maximum number of samples used in estimation the same as those in Table 14.8.

For Kakula, SG data were available for the majority of drillhole samples, and regression values were available where sample data were missing (Section 14.5). SG was estimated using ID3 and the same search parameters used for TCu, but was not weighted by TT.

14.10 Dilution Skins

Given the full 3D nature of the Kamoa model, no dilution skins were applied, as dilution grades were available from bounding domains. For Kakula, dilution skins 0.3 m thick were applied to the top (hangingwall) and bottom (footwall) contacts of the SMZ. Grades were estimated into these blocks using only composites that occurred within the 0.3 m interval. The dilution skins were constructed to allow mining studies to use an estimated dilution grade relevant to the material being added.

14.11 Mineral Resource Classification

The same drillhole spacing criteria are used at both Kamoa and Kakula to classify Mineral Resources. Areas outlined by core drilling at 800 m spacing with a maximum projection distance of 600 m outward of drill sections, and which show continuity of grade at 1% TCu, geological continuity, and continuity of structure (broad anticline with local discontinuities that are likely faults) were classified as Inferred Mineral Resources over a combined area of 24.9 km^2. Mineral Resources within a combined area of 70.1 km^2 that were drilled on 400 m spacing and which display grade and geological continuity were classified as Indicated Mineral Resources. The total area of the Kamoa-Kakula Project is approximately 410.1 km^2.

The resource classification for Kamoa is shown in Figure 14.23, and for Kakula in Figure 14.24. Although the Kamoa resource model was updated from 2014, all additional drillholes were infill holes within the Indicated outline. Only minor changes to account for adjustments to the shape of the domes have been made to the classification since 2013.

Figure 14.23 Kamoa: Mineral Resource Classification

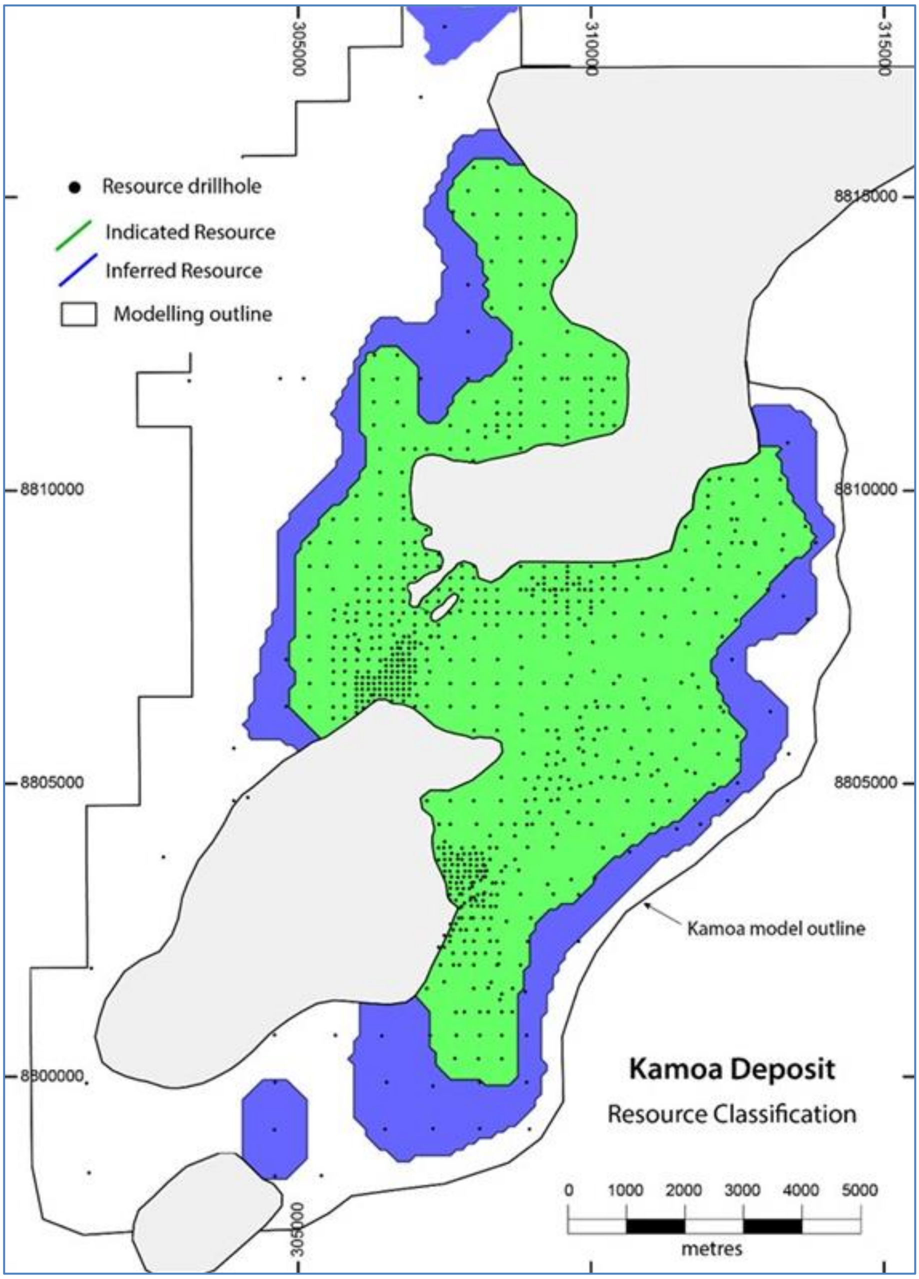


Figure prepared by Ivanhoe, 2017.

Figure 14.24 Kakula: Mineral Resource Classification and Expansion Since 2017

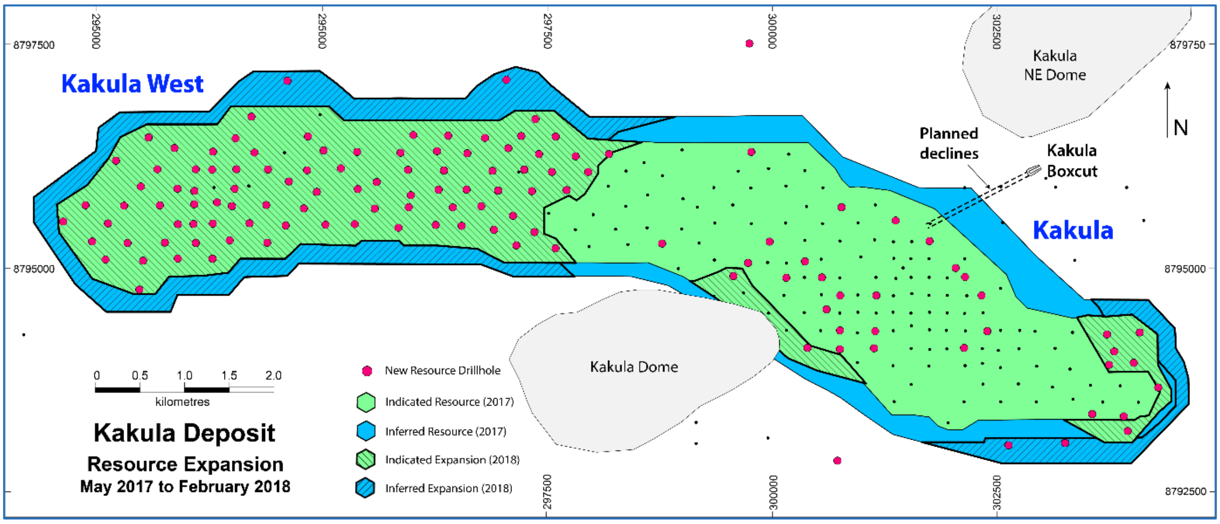


Figure provided by Ivanhoe, 2018.

14.12 Model Validations

14.12.1 Visual Checks

Estimated block grades and composite grades were compared visually in plan view and showed a good agreement, refer to Figure 14.25 for Kamoa, and Figure 14.26 and Figure 14.27 for the 3% grade shell at Kakula.

The previous 2D modelling approach used at Kamoa was updated for the additional drillhole data to allow comparison of the 2D and 3D models. To ensure a fair comparison, the 2D model was trimmed within the Upper SMZ perimeter, and the 3D model blocks were selected within a wireframe defining the upper and lower contacts of the 1.0% 2D model (SMZ10). These blocks were combined to the 2D model block size of 100 m x 100 m (i.e. sets of four 50 m x 50 m blocks were combined together) and a single cell in the vertical representing the vertical thickness.

These models were compared visually and graphically through plotting the grade and tonnage at different cut-offs (Figure 14.28), and by plotting the grade distributions in plan view (Figure 14.29). These comparisons indicate that the SMZ selection and estimation technique have both contributed to a smoothing of the block grades in the 3D model when considered over the full mineralised interval; however, this smoothing is overcome when the vertical definition available in the original 50 m x 50 m x 1 m blocks from the 3D model are used (the stippled lines in Figure 14.28).

Figure 14.25 Kamoa: Estimated TCu Grade (%) for the Upper SMZ (Domain 300)

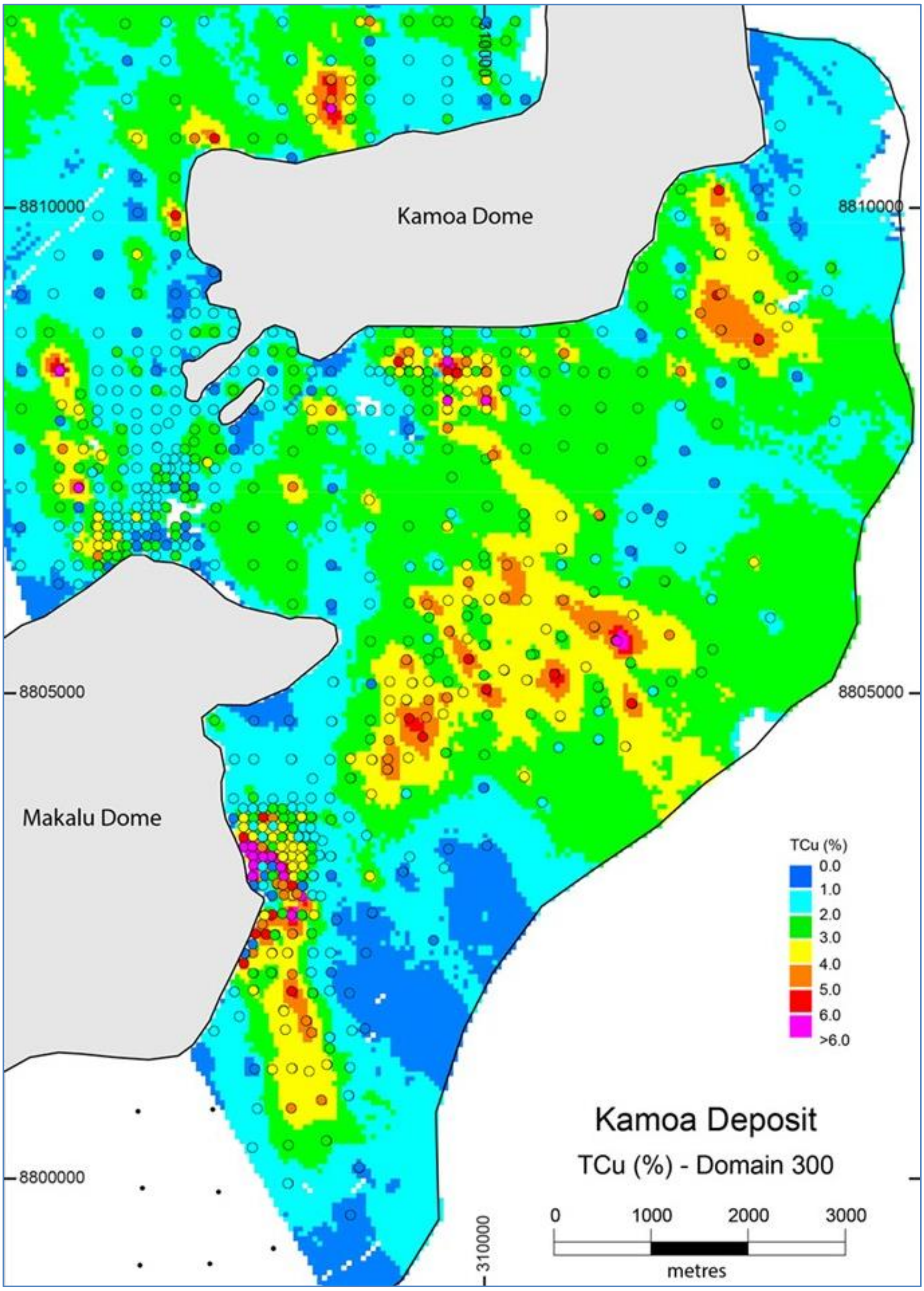


Figure prepared by Ivanhoe, 2017.

Figure 14.26 Kakula: Estimated TCu (%) for the 3.0% TCu Grade Shell (SMZ30)

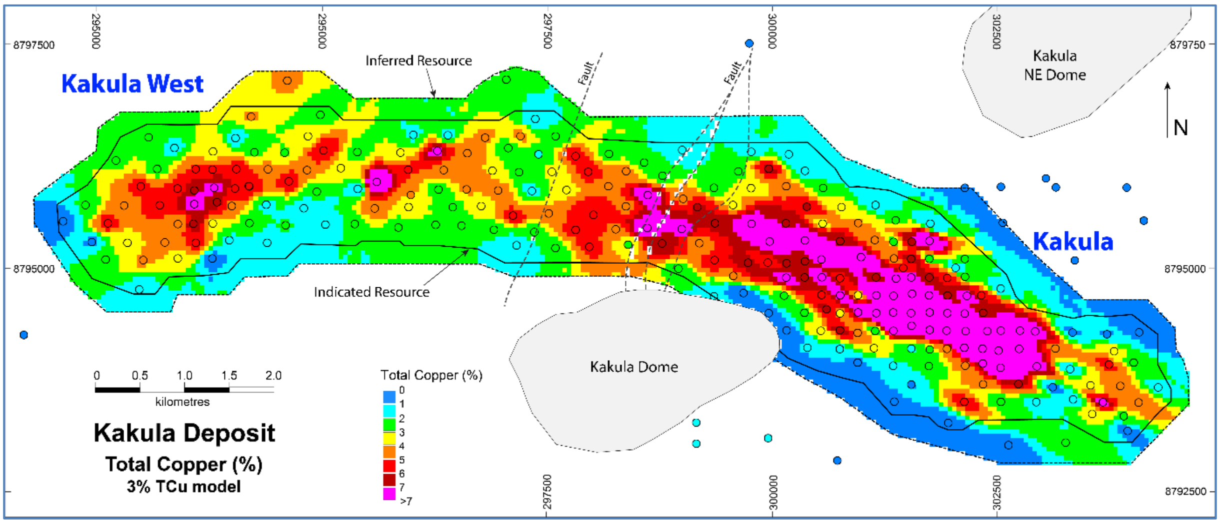


Figure prepared provided by Ivanhoe, 2018. Missing blocks are due to fault offsets.

Figure 14.27 Kakula: Estimated True Thickness (m) for the 3.0% TCu Grade Shell (SMZ30)

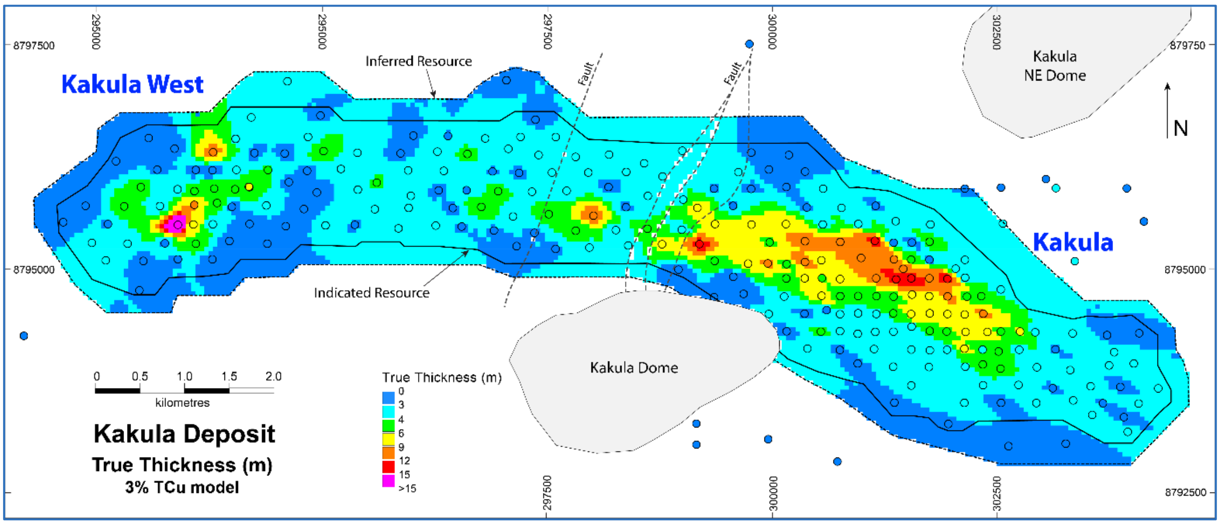


Figure prepared by Ivanhoe, 2018. Missing blocks are due to fault offsets.

Figure 14.28 Comparative Grade-Tonnage Curves for the Kamoa 2D and 3D Models Constrained Within the Upper SMZ (Domains 110, 300, and 310) and Wireframe Defining the 1.0% TCu 2D Model (SMZ10)

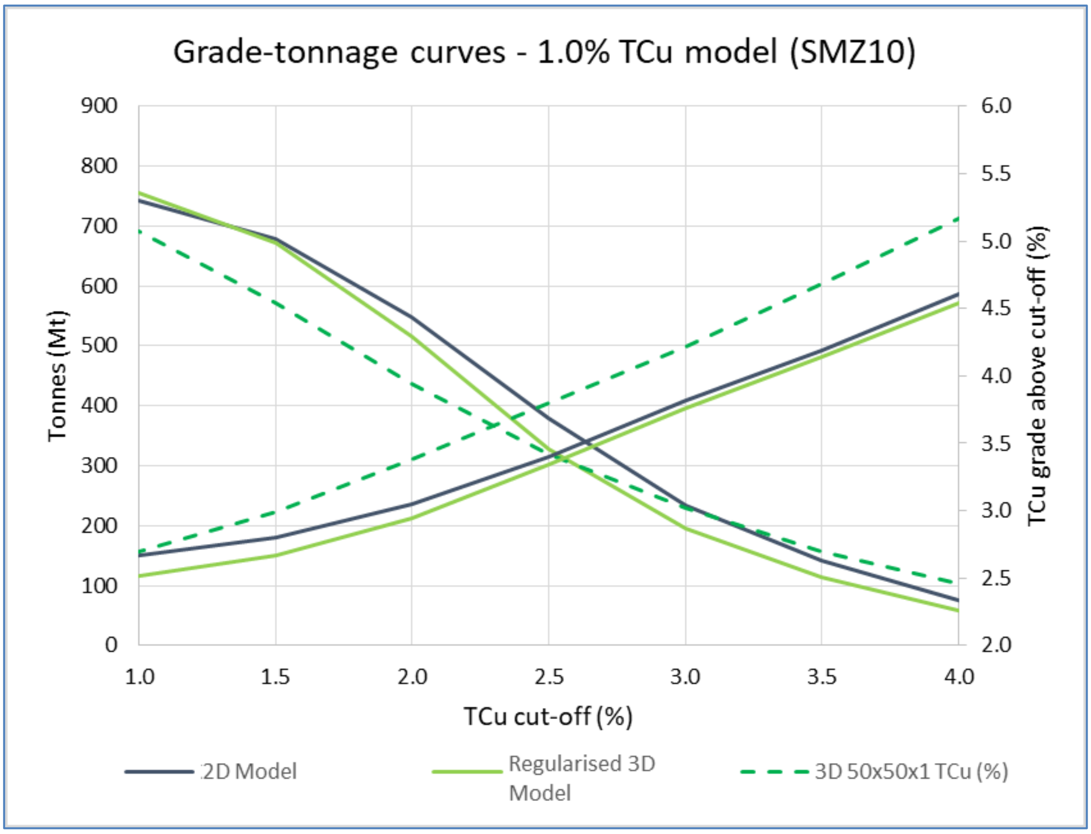


Figure prepared by Ivanhoe, 2018. Solid lines represent two-dimensional model (100 m x 100 m x variable height) and two-dimensional equivalent of the 3D model ('Regularised 3D Model'); stippled line represents reporting from the 3D model (50 m x 50 m x 1 m).

Figure 14.29 Comparison of 3D (left) and 2D (right) Models for Estimated TCu Grade (%) for a 1.0% TCU Modelling Cut-off (SMZ10)

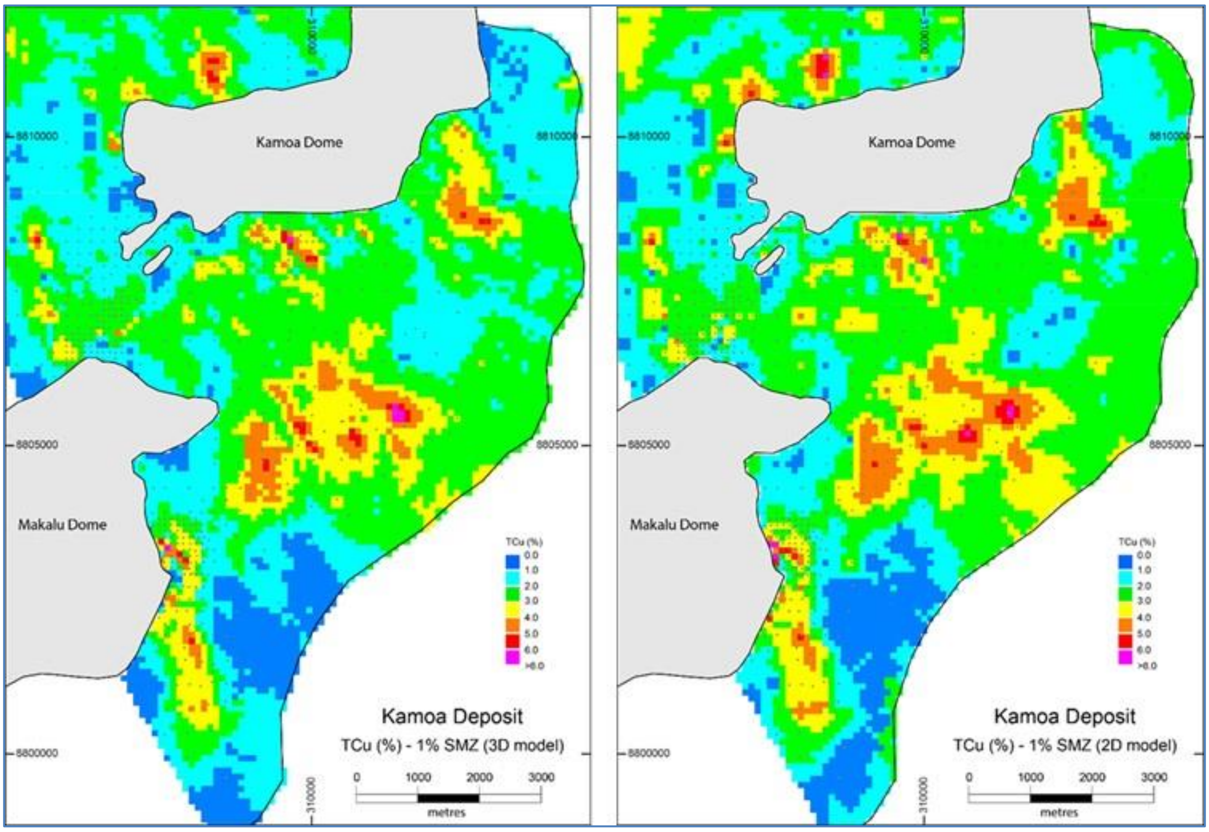


Figure prepared provided by Ivanhoe, 2017, black dots are drillholes.

14.13 Global Bias Checks

14.13.1 Global Bias

During estimation, a declustered drillhole or NN estimate was included to allow a check for global bias between the estimated grade and the drillhole grades. The NN estimate minimises the effect of clustering of data and allows for a more appropriate comparison with estimated grades. This is particularly relevant at Kamoa, where a significant clustering of data occurs in the shallower portions of the deposit, close to the dome edges. Relative differences between the ID and NN models are generally below 5%, which is considered appropriate for an Indicated classification. Global biases are summarised in Table 14.11 for Kamoa, and in Table 14.12 using the key 3% SMZ30 model at Kakula.

Table 14.11 Kamoa: Mean Grades for 1.0% Cut-off (SMZ10) Composites and Models

Indicated (no cut-off applied)	Composite	Model (OK)	Model (NN)	Relative Diff (OK-NN)/NN
TCu (%)	2.56	2.35	2.28	3.1%
S (%)	1.60	1.66	1.69	-1.8%

Table 14.12 Kakula: Mean Grades for 3.0% Cut-off (SMZ30) Composites and Models

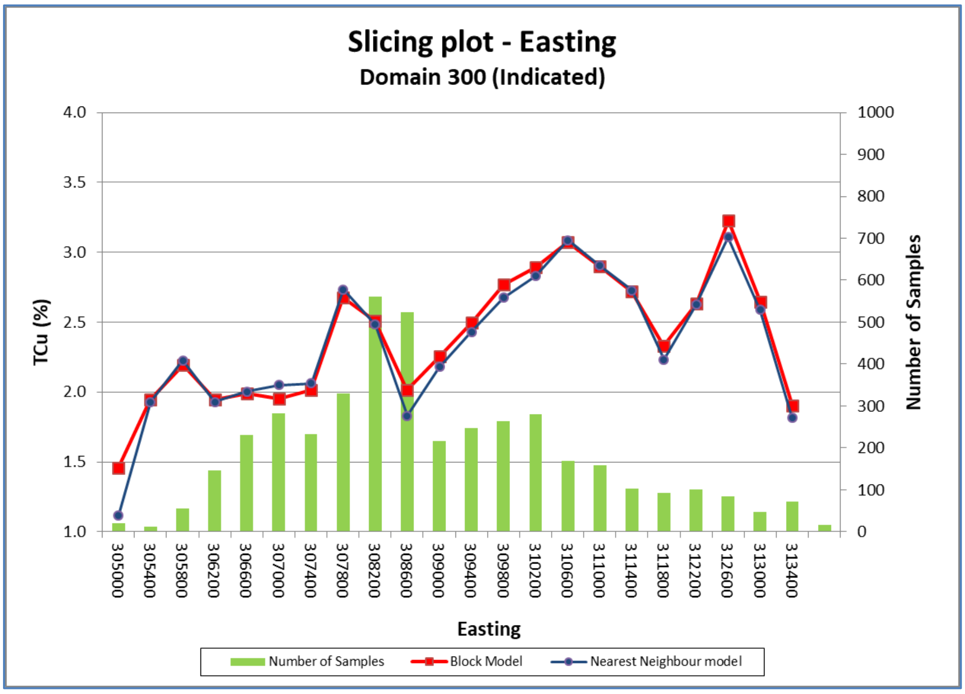
Indicated (No cut-off applied)	Composite	Model (ID3)	Model (NN)	Relative Diff. (ID3-NN)/NN
TCu * TT * SG	64.62	52.02	51.33	1.3%
True Thickness (TT)	4.43	4.05	4.03	0.5%
SG	2.82	2.81	2.81	0.0%
Inferred				
TCu * TT * SG	16.30	16.33	14.13	15.6%
True Thickness (TT)	2.96	3.03	2.98	1.7%
SG	2.79	2.78	2.78	0.1%

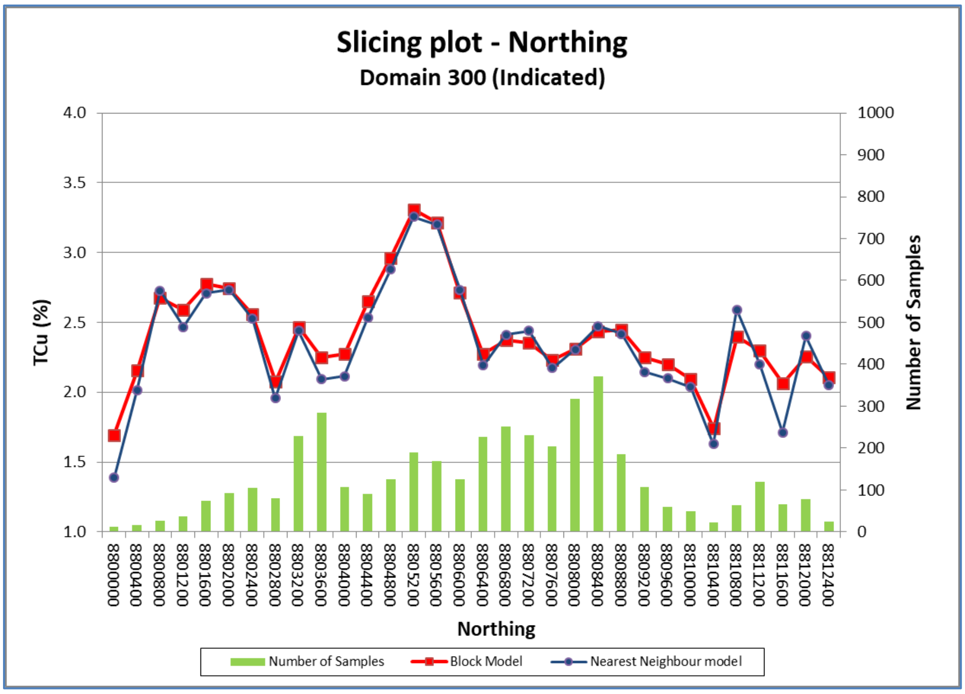
14.13.2 Local Bias Checks (Swath or Slicing Plots)

14.13.2.1 Kamoa

Checks for local bias were performed for TCu by analysing local grade trends on 400 m slices (swaths) in easting and northing. Example swath plots for TCu (%) are shown in Figure 14.30. The average grade per swath for the block model (red line) is compared with the average grade for the declustered drillholes (represented by the NN estimate) (blue line) for the same swath. The two lines are observed to follow very similar trends, indicating that no local biases are evident.

Figure 14.30 Kamoa: Swath Plots for TCu (%) for the Upper SMZ (Domain 300)





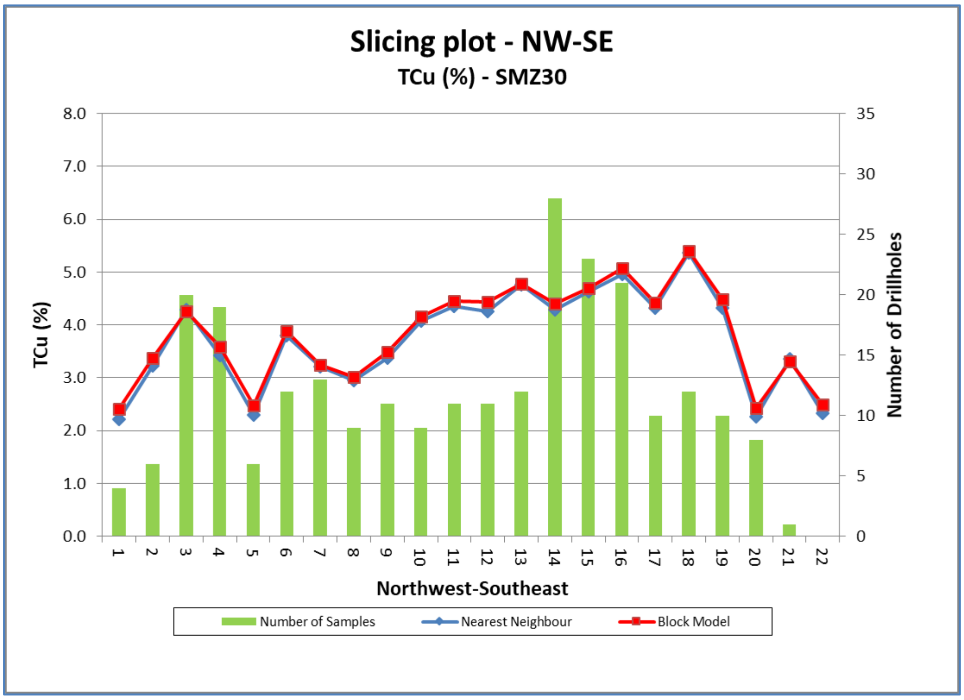
Figures prepared by Ivanhoe, 2018.

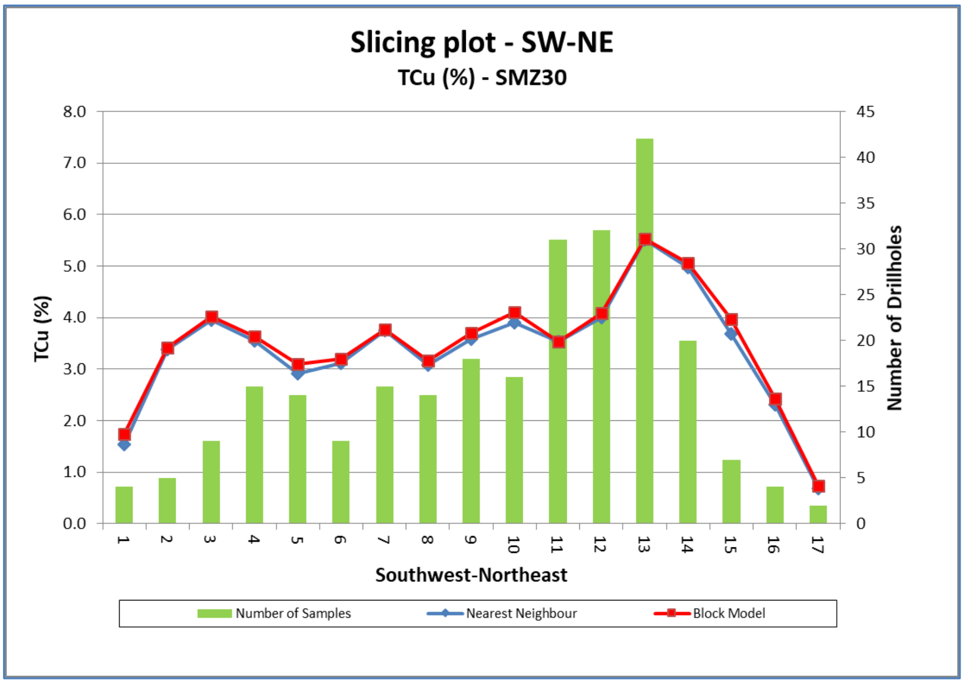
14.13.2.2 Kakula

Checks for local bias were performed for TCu, TT, TCuxTTxSG, and SGxTT by analysing local grade trends on 500 m swaths aligned north-west south-east (along the trend of the high-grade mineralisation), and 500 m swaths aligned south-west north-east (across the trend of the high-grade mineralisation).

Example swath plots for TCu (%) and true thickness (m) are shown for the key SMZ30 model in Figure 14.31 and Figure 14.32, respectively. The average grade per swath for the block model (red line) is compared with the average grade for the declustered drillholes (represented by the NN estimate) (blue line) for the same swath. The two lines are observed to follow very similar trends, indicating that no local biases are evident.

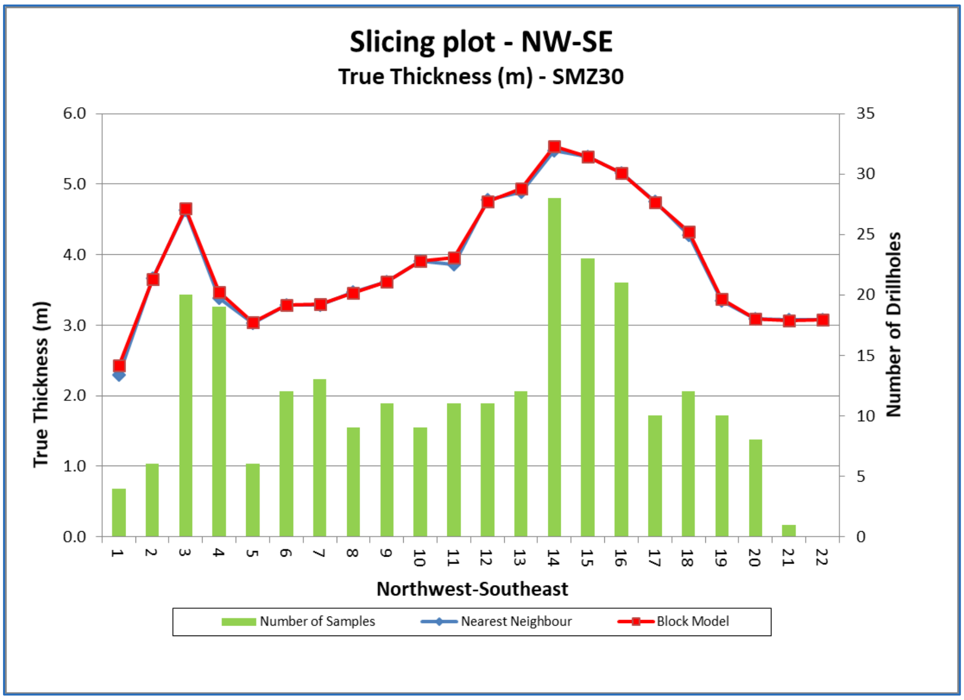
Figure 14.31 Kakula: Swath Plots for TCu (%) for the 3.0% TCu Grade Shell (SMZ30)

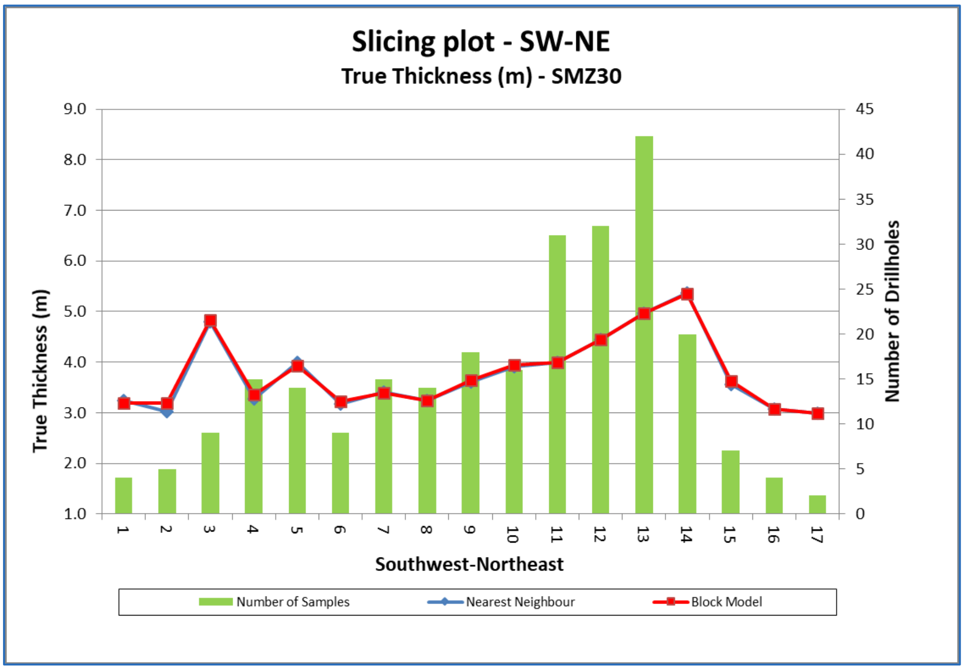




Figures prepared provided by Ivanhoe, 2018.

Figure 14.32 Kakula: Swath Plots for True Thickness (m) for the 3.0% TCu Grade Shell (SMZ30)





Figures prepared provided by Ivanhoe, 2018.

14.14 Reasonable Prospects of Eventual Economic Extraction

Amec Foster Wheeler has used a 1% TCu cut-off grade to support Mineral Resource estimation. This choice of cut-off is based on many years of mining experience on the Zambian Copperbelt at mines such as Konkola, Nchanga, Nkana, and Luanshya, which mine similar mineralisation to that identified at Kamoa and Kakula.

14.14.1 Kamoa Assessment of Reasonable Prospects for Eventual Economic Extraction

To test the cut-off grade for the purposes of assessing reasonable prospects of eventual economic extraction, Amec Foster Wheeler performed a conceptual analysis based on conditions considered appropriate for the region. A copper price of US$3.00/lb was assumed. The following additional key parameters were used:

- Percent recovery for hypogene is based on a reference case having a feed grade of 3.54% Cu and a tailings grade of 0.44% Cu. The reference case gives a recovery of 88.7%, and the adjusted recovery is 77.3% at a feed grade of 1.0% Cu.
- Percent recovery for supergene is based on a reference case having a feed grade of 3.54% Cu, and a tailings grade of 0.43% Cu. The reference case gives a metallurgical recovery of 88.7%. If the ASCu/TCu ratio is ≤ 0.125, the block is treated using hypogene recovery equations, but with an assumed concentrate grade of 45% TCu. If the ASCu/TCU ratio is > 0.125, the sulphide copper is estimated to be (TCu-ASCu) + 0.125 TCu. The hypogene recovery equation is then applied to sulphide copper, but with an assumed concentrate grade of 45% TCu. Estimated TCu recovery at a 1% TCu feed grade is 77.5% where the ASCu/TCu ratio is ≤ 0.125. Estimated TCu recovery at a 1% TCu feed grade where the ASCu/TCU ratio is 0.30 is 74.2%.
- Concentrate grades for supergene of 45.0% TCu and 16.9% S.
- Concentrate grades for hypogene of 36.0% TCu and 31.6% S.
- Concentrate moisture of 12%.
- Mining costs of US$27/t.
- Concentrator, tailings treatment and G&A costs of US$17/t treated.
- Payable copper of 97.1% for the supergene case and 96.4% for the hypogene case.
- Smelting costs of US$80/t of concentrates.
- Refining costs of US$0.08/lb payable copper.
- Transport costs of concentrates to smelter US$323/wmt concentrates.
- Royalty of 2% on payable copper – smelting costs – refining – transport costs.
- National Export Tax of 1% of payable copper – smelting costs – refining costs.
- Concentrate tax of US$100/wmt concentrates.
- NSR = payable copper – smelting costs – refining costs – transport costs – royalties – taxes.

The hypogene metallurgical recovery equation is based on the updated curve shown in Figure 13.14. The supergene metallurgical recovery equations are based on an analysis of the ASCu/TCu ratios of material designated as supergene:

- This analysis shows that 54% of the blocks classified as supergene have similar ASCu/TCu ratios as hypogene blocks. These blocks have ratios less than 0.125 and an average ratio of 0.084. These blocks (51% of all supergene blocks) are termed Group A.
- Where the ratio is > 0.125, their sulphide (floatable) copper is assumed to be (TCu-ASCu +0.125TCu). These blocks (49% of all supergene blocks) are termed Group B.

Group A blocks have metallurgical recoveries estimated using TCu and the hypogene equations. Group B blocks have recoveries estimated using sulphide copper as input to the hypogene equations. Figure 14.34 shows the % TCu recovery versus TCu feed grade. The overall TCu recovery for groups A+B is about 3.7% lower than would be obtained if the hypogene equation (Group A) were used throughout. The graph shows that the curve (Figure 13.14) based on Phase 6 ISF4a is likely based on a non-representative supergene sample with abundant copper oxides.

Figure 14.33 % TCu Recovery Versus % TCu in Feed for Kamoa Supergene Blocks

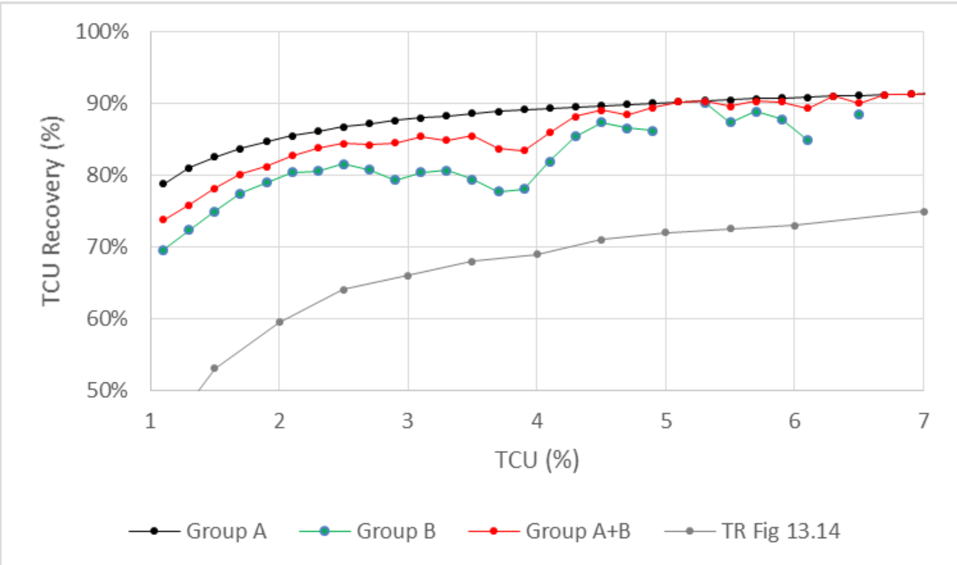


Figure Amec Foster Wheeler, 2018.

Normally, cut-off grades used to declare Mineral Resources do not consider mining costs. There are additional areas for which reasonable prospects for eventual economic extraction exist and which might be scheduled if the nominal 12 Mtpa production rate used for the Kamoa-Kakula 2017 PEA was increased to as much as 20 Mtpa. These additional areas are included using a 1% TCu cut-off. There is a small percentage (~4 %) of the tonnage representing 2% of the contained copper that has copper grades between 1.0% and 1.25%; the NSRs for these blocks will cover onsite concentrator, tailings treatment, and G&A costs but will not cover their full mining costs. Blocks grading between 1.0% and 1.25% TCu are estimated to cover $23/t of $27/t assumed mining costs. It may be convenient to mine these blocks in conjunction with adjacent higher-grade blocks, and therefore Amec Foster Wheeler has included the blocks in the Mineral Resource tabulations. Based on these assumptions, the Mineral Resources are considered to have met the requirement for reasonable prospects for eventual economic extraction.

As a sensitivity analysis, Amec Foster Wheeler considered a case in which an on-site smelter would produce blister copper (~99% Cu), as savings would be realised in terms of reduced transport of product costs. In addition, sulphuric acid of 98.5% purity would be produced for sale using a price of US$200/t. This is perhaps a more realistic case in that the Kamoa resource base is large enough to contemplate on-site smelting (as was done for the 2013 PEA). For this case the NSRs for all blocks meeting a 1% Cu cut-off grade would cover onsite processing, tailings treatment, and G&A costs. There is a small percentage (0.4% of the tonnage and 0.2% of the metal) of blocks that will not cover full mining costs. These blocks will cover $18/t of $27/t assumed mining costs.

Amec Foster Wheeler cautions that with the underground mining methods envisioned (room-and-pillar or drift-and-fill), the mining recovery may vary from 55% to 80% depending on the success in which pillars can be mined on retreat, and/or a backfill or convergence method is used. In addition, the Mineral Resources do not incorporate allowances for contact (external) dilution at the roof and floor of the deposit. This will ultimately depend on the ability of the mining operation to follow the SMZ boundaries.

14.14.2 Kakula Assessment of Reasonable Prospects for Eventual Economic Extraction

In 2017 Amec Foster Wheeler applied the same methodology used for Kamoa to assess reasonable prospects for eventual economic extraction for Kakula (see Section 14.14.1). The assumptions incorporate a copper price of US$3.00/lb. It was found that the NSR (as defined below) for all Mineral Resources at a cut-off of 1% TCu will cover processing, tailings treatment, and general and administrative (G&A) costs. However, blocks grading between 1% and 1.45% TCu will not cover full mining costs. It may be convenient to mine these blocks in conjunction with adjacent higher-grade blocks. These blocks represent 4% of the Mineral Resource tonnage and 2% of the contained copper. Based on this analysis, Amec Foster Wheeler considers the 2017 Mineral Resource estimates to be current for the purposes of this Technical Report. The assumptions made in Section 14.14.1 broadly apply to Kakula. Minor changes have been made as follows:

- Concentrate grade of 54.9% TCu.
- Concentrator metallurgical recoveries range from 78% at a 1.0% TCu grade to 85% at the average grade of the Indicated Mineral Resource.
- Concentrate moisture of 12%.
- Mining costs of US$42/t.
- Concentrator, tailings treatment and G&A costs of US$18/t treated.
- Payable copper of 97.6%.
- Smelting costs of US$80/t of concentrates.
- Refining costs of US$0.08/lb payable copper.
- Transport costs of concentrates to smelter US$323/wmt concentrates.
- Royalty of 2% on payable copper – smelting costs – refining costs – transport.
- National Export Tax of 1% of payable copper – smelting costs – refining costs.
- Concentrate tax of US$100/wmt concentrates.
- NSR = payable copper – smelting costs – refining costs – transport costs – royalties – taxes.

All mineralised material at Kakula is considered to be hypogene and is based on a reference case having a feed grade of 6.01% Cu and a tailings grade of 0.86% Cu. The reference case gives a metallurgical recovery of 87.1%, and the adjusted recovery is 77.7% at a feed grade of 1.0% Cu.

There are reasonable prospects for eventual economic extraction under assumptions of a copper price of US$3.00/lb, employment of underground mechanised room-and-pillar and drift-and-fill mining methods, and that copper concentrates will be produced and sold to a smelter. At a 1% TCu cut-off grade, the assumed NSRs for 100% of Mineral Resource blocks will cover concentrating, tailings treatment, and G&A costs.

As at Kamoa, there is a proportion (25%) of the tonnage representing only 11% of the contained copper in the Mineral Resource at Kakula that will not cover its full mining costs; e.g. blocks grading between 1% and 1.45% TCu. It may be convenient to mine these blocks in conjunction with adjacent higher-grade blocks, and therefore Amec Foster Wheeler has included the blocks in the Mineral Resource tabulations. For example, blocks grading between 1% and 1.45% TCu will have an average grade of 1.28% TCu, and these will cover $34/t out of the assumed $42/t mining costs. Based on these assumptions, the Mineral Resources are considered to have met the requirement for reasonable prospects for eventual economic extraction.

As at Kamoa, Amec Foster Wheeler evaluated an on-site smelting case and found the reduction in operating costs, transportation and taxes allowed for blocks meeting a 1.25% TCu cut-off grade to cover concentrator, tailings treatment, G&A, and full mining costs. There are 9% of the blocks containing 4% of the contained copper between cut-offs of 1% and 1.25% TCu that will cover concentrator, tailings treatment, G&A and some mining costs (average $38/t of $42/t assumed total mining costs).

14.15 Mineral Resource Statement

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

14.15.1 Kamoa Mineral Resource Statement

Indicated and Inferred Mineral Resources for the 3D resource model are summarised in Table 14.13. Mineral Resources are reported inclusive of Mineral Reserves on a 100% basis.

The Mineral Resources for Kamoa have an effective date of 27 November 2017. The Mineral Resources do not include any material in the hangingwall and footwall, and make no allowance for mining recovery factors. To avoid reporting isolated blocks above cut-off, a minimum stack of three contiguous vertical blocks (3 m vertical thickness) was required to meet the cut-off criteria for the tonnage and grade estimate to be reported. In addition, where two or more distinct mineralised zones occurred in the same vertical profile, only the highest metal content zone was reported if the secondary mineralised zone could not justify the dilution between the two zones and still remain above cut-off over the combined interval.

Table 14.13 Kamoa Indicated and Inferred Mineral Resource (at 1% TCu Cut-off Grade)

Category	Tonnage (Mt)	Area (km^2)	Copper (%)	Vertical Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
Indicated	759	50.7	2.57	5.5	19,500	43.0
Inferred	202	19.4	1.85	3.8	3,740	8.2

7. Ivanhoe's Mineral Resources Manager George Gilchrist, a Fellow of the Geology Society of South Africa and Professional Natural Scientist (Pr. Sci. Nat) with the South African Council for Natural Scientific Professions (SACNASP), estimated the Mineral Resources under the supervision of Dr. Harry Parker and Gordon Seibel, both Registered Members (RM) of the Society for Mining, Metallurgy and Exploration (SME), employees of Amec Foster Wheeler, who are the Qualified Persons for the Mineral Resource estimate. The effective date of the estimate is 27 November 2017 and the cut-off date for drill data is 23 November 2015. Mineral Resources are estimated using the CIM 2014 Definition Standards for Mineral Resources and Mineral Reserves. Mineral Resources are reported inclusive of Mineral Reserves on a 100% basis.
8. Mineral Resources are reported using a total copper (TCu) cut-off grade of 1% TCu and a minimum vertical thickness of 3 m. There are reasonable prospects for eventual economic extraction under assumptions of a copper price of US$3.00/lb, employment of underground mechanised room-and-pillar and drift-and-fill mining methods, and that copper concentrates will be produced and sold to a smelter. Mining costs are assumed to be US$27/t. Concentrator, tailings treatment, and general and administrative costs (G&A) are assumed to be US$17/t. Metallurgical recoveries are expected to average 84% (86% forhypogene and 81% forsupergene). At a 1% TCu cut-off grade, assumed net smelter returns for 100% of Mineral Resource blocks will cover processing, tailings treatment and G&A costs.
9. Reported Mineral Resources contain no allowances for hangingwall or footwall contact boundary loss and dilution. No mining recovery has been applied.
10. Depth of mineralisation below the surface ranges from 10 m to 1,320 m for Indicated Mineral Resources and 20 m to 1,560 m for Inferred Mineral Resources.
11. Approximate drillhole spacings are 800 m for Inferred Mineral Resources and 400 m for Indicated Mineral Resources.
12. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.

14.15.2 Kakula Mineral Resource Statement

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

Indicated and Inferred Mineral Resources for Kakula have an effective date of 23 February 2018 and are summarised in Table 14.14 on a 100% basis. No Mineral Reserves have been estimated for Kakula. The Mineral Resources do not include any material in the hangingwall and footwall dilution skins and make no allowance for mining recovery factors. All units are reported using metric units, with the exception of contained copper pounds, which is reported in imperial units.

At any reporting cut-off, but specifically relevant at cut-offs <3% TCu, blocks from the 1% and 2% TCu grade shells are only considered in the reporting if the 3% TCu grade shell at the same easting and northing location is above the cut-off grade.

Table 14.14 Kakula: Indicated and Inferred Mineral Resource (at 1% TCu Cut-off Grade)

Category	Tonnage (Mt)	Area (km^2)	Copper (%)	Vertical Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
Indicated	585	19.4	2.92	10.8	17,100	37.7
Inferred	113	5.5	1.90	7.3	2,150	4.7

1. Ivanhoe's Mineral Resources Manager George Gilchrist, a Fellow of the Geology Society of South Africa and Professional Natural Scientist (Pr. Sci. Nat) with the South African Council for Natural Scientific Professions (SACNASP), estimated the Mineral Resources under the supervision of Dr. Harry Parker and Gordon Seibel, both Registered Members (RM) of the Society for Mining, Metallurgy and Exploration (SME), employees of Amec Foster Wheeler, who are the Qualified Persons for the Mineral Resources. The effective date of the estimate is 23 February 2018, and the cut-off date for the drill data is 26 January 2018. Mineral Resources are estimated using the CIM Definition Standards for Mineral Resources and Reserves (2014) and reported on a 100% basis. No Mineral Reserves are reported at Kakula.
2. Mineral Resources are reported using a total copper (TCu) cut-off grade of 1% TCu and an approximate minimum thickness of 3 m. There are reasonable prospects for eventual economic extraction under assumptions of a copper price of US$3.00/lb, employment of underground, mechanised, room-and-pillar and drift-and-fill mining methods, and that copper concentrates will be produced and sold to a smelter. Mining costs are assumed to be US$42/t. Concentrator, tailings treatment and general and administrative (G&A) costs are assumed to be US$18/t. Metallurgical recovery is assumed to average 85%. Ivanhoe is studying reducing mining costs using a controlled convergence room-and-pillar method. At a 1% TCu cut-off grade, assumed net smelter returns for 100% of Mineral Resource blocks will cover concentrator, tailings treatment and G&A costs.
3. Reported Mineral Resources contain no allowances for hangingwall or footwall contact boundary loss and dilution. No mining recovery has been applied.
4. Approximate drillhole spacings are 800 m for Inferred Mineral Resources and 400 m for Indicated Mineral Resources.
5. Rounding as required by reporting guidelines may result in apparent differences between tonnes, grade and contained metal content.

14.15.3 Kamoa-Kakula Project

Indicated and Inferred Mineral Resources for the Kamoa-Kakula Project are provided on a 100% basis in Table 14.15. The Mineral Resources in Table 14.13 and Table 14.14 are not additive to this table.

Table 14.15 Kamoa and Kakula: Indicated and Inferred Mineral Resource (at 1% TCu Cut-off Grade)

Deposit	Category	Tonnes (millions)	Area (Sq. km)	Copper Grade (%)	Vertical Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
Kamoa	Indicated	759	50.7	2.57	5.5	19,500	43.0
	Inferred	202	19.4	1.85	3.8	3,740	8.2
Kakula	Indicated	585	19.4	2.92	10.8	17,100	37.7
	Inferred	113	5.5	1.90	7.3	2,150	4.7
Total Kamoa-Kakula Project	Indicated	1,340	70.1	2.72	6.9	36,600	80.7
	Inferred	315	24.9	1.87	4.6	5,890	13.0

6. Ivanhoe's Mineral Resources Manager, George Gilchrist, Professional Natural Scientist (Pr. Sci. Nat) with the South African Council for Natural Scientific Professions (SACNASP), estimated the Mineral Resources under the supervision of Dr. Harry Parker and Gordon Seibel, both Registered Members (RM) of the Society for Mining, Metallurgy and Exploration (SME), who are the Qualified Persons for the Mineral Resource estimate. The effective date of the estimate is 23 February 2018, and the cut-off date for the drill data is 26 January 2018. Mineral Resources are estimated using the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves. Mineral Resources at Kamoa are inclusive of Mineral Reserves. No Mineral Reserves are currently reported at Kakula.
7. Mineral Resources are reported for Kamoa using a total copper (TCu) cut-off grade of 1% TCu and a minimum vertical thickness of 3 m. There are reasonable prospects for eventual economic extraction under assumptions of a copper price of US$3.00/lb, employment of underground mechanised room-and-pillar and drift-and-fill mining methods, and that copper concentrates will be produced and sold to a smelter. Mining costs are assumed to be US$27/t. Concentrator, tailings treatment, and general and administrative (G&A) costs are assumed to be US$17/t. Metallurgical recovery will average 84% (86% forhypogene and 81% forsupergene). At a 1% TCu cut-off grade, assumed net smelter returns for 100% of Mineral Resource blocks will cover concentrator, tailings treatment and G&A costs.
8. Mineral Resources are reported for Kakula using a TCu cut-off grade of 1% TCu and an approximate minimum thickness of 3 m. There are reasonable prospects for eventual economic extraction under assumptions of a copper price of US$3.00/lb, employment of underground, mechanised, room-and-pillar and drift-and-fill mining methods, and that copper concentrates will be produced and sold to a smelter. Mining costs are assumed to be US$42/t. Concentrator, tailings treatment and G&A costs are assumed to be US$18/t. Metallurgical recovery is assumed to average 85%. Ivanhoe is studying reducing mining costs using a controlled convergence room-and-pillar method. At a 1% TCu cut-off grade, assumed net smelter returns for 100% of Mineral Resource blocks will cover concentrator, tailings treatment and G&A costs.
9. Reported Mineral Resources contain no allowances for hangingwall or footwall contact boundary loss and dilution. No mining recovery has been applied.
10. Approximate drillhole spacings are 800 m for Inferred Mineral Resources and 400 m for Indicated Mineral Resources.
11. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.The Mineral Resources in Table 14.13 and Table 14.14 are not additive to this table.

14.16 Sensitivity of Mineral Resources to Cut-off Grade

Table 14.16 summarises the Kamoa Mineral Resource at a range of cut-off grades. The base case Mineral Resource model reported at a 1.0% TCu cut-off is highlighted in grey.

Table 14.16 Kamoa: Sensitivity of Mineral Resources to Cut-off Grade

Indicated Mineral Resource						
Cut-off (% Cu)	Tonnage (Mt)	Area (km^2)	Copper (%)	Vertical Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
3.00	223	20.9	4.14	3.8	9,230	20.4
2.50	328	29.5	3.70	4.0	12,100	26.7
2.00	450	37.2	3.30	4.4	14,900	32.8
1.75	525	41.2	3.10	4.6	16,300	35.9
1.50	607	44.8	2.90	4.9	17,600	38.8
1.25	683	47.8	2.73	5.2	18,700	41.1
1.00	759	50.7	2.57	5.5	19,500	43.0
0.75	849	52.9	2.39	5.9	20,300	44.7
Inferred Mineral Resource						
3.00	19	2.1	3.52	3.2	655	1.4
2.50	40	4.4	3.10	3.3	1,250	2.8
2.00	67	7.1	2.77	3.4	1,840	4.1
1.75	88	9.2	2.55	3.5	2,240	4.9
1.50	114	11.8	2.34	3.5	2,660	5.9
1.25	151	15.3	2.10	3.6	3,170	7.0
1.00	202	19.4	1.85	3.8	3,740	8.2
0.75	253	22.5	1.65	4.1	4,180	9.2

The footnotes to Table 14.13 also apply to this table.

Table 14.17 summarises the Kakula Mineral Resource at a range of cut-off grades. The base case Mineral Resource model reported at a 1.0% TCu cut-off is highlighted in grey.

Table 14.17 Kakula: Sensitivity of Mineral Resources to Cut-off Grade

Indicated Mineral Resource						
Cut-off (% Cu)	Tonnage (Mt)	Area (km^2)	Copper (%)	True Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
7.0	41	2.2	8.07	6.3	3,290	7.3
6.0	67	3.6	7.46	6.2	4,970	11.0
5.0	98	5.7	6.82	5.7	6,690	14.7
4.0	140	9.0	6.13	5.1	8,560	18.9
3.0	174	12.3	5.62	4.7	9,750	21.5
2.5	208	14.4	5.14	4.8	10,700	23.5
2.0	330	16.6	4.07	6.6	13,400	29.6
1.5	420	18.0	3.55	7.8	14,900	32.9
1.0	585	19.4	2.92	10.1	17,100	37.7
Inferred Mineral Resource						
4.0	2	0.2	4.17	3.3	98	0.2
3.0	9	0.8	3.66	3.3	325	0.7
2.5	17	1.7	3.20	3.2	549	1.2
2.0	44	3.2	2.59	4.3	1,140	2.5
1.5	69	4.5	2.26	5.0	1,560	3.4
1.0	113	5.5	1.90	6.7	2,150	4.7

The footnotes to Table 14.14 also apply to this table.

Table 14.18 summarises the Kamoa-Kakula Project Mineral Resource estimate at a range of cut-off grades. The base case Mineral Resource model reported at a 1.0% TCu cut-off is highlighted in grey.

Table 14.18 Kamoa and Kakula: Sensitivity of Project Mineral Resources to Cut-off Grade

Indicated Mineral Resource						
Cut-off (% Cu)	Tonnage (Mt)	Area (km²)	Copper (%)	Vertical Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
3.0	396	33.2	4.79	4.2	19,000	41.8
2.5	535	44.0	4.25	4.4	22,800	50.2
2.0	780	53.8	3.63	5.2	28,300	62.4
1.5	1,030	62.8	3.17	5.9	32,500	71.7
1.0	1,340	70.1	2.72	6.9	36,600	80.7
Inferred Mineral Resource						
3.0	28	3.0	3.56	3.3	979	2.2
2.5	58	6.1	3.13	3.3	1,800	4.0
2.0	111	10.3	2.69	3.9	2,980	6.6
1.5	183	16.3	2.31	4.0	4,220	9.3
1.0	315	24.9	1.87	4.6	5,890	13.0

The footnotes to Table 14.15 also apply to this table.

14.17 Considerations for Mine Planning

The Kamoa deposit poses a significant challenge to building a reliable 3D model due to the deposit's lateral extent of tens of kilometres, and a vertical mineralisation extent of a few metres. These challenges, however, are minimised by the significant amount of high-quality drillhole data and the general consistency and predictability of the mineralisation.

Kamoa has traditionally been modelled using a 2D approach at a defined cut-off, or at a series of defined cut-offs. By averaging the grades over the full vertical extent of the SMZ, the vertical height of the mineralisation was fixed.

The 2017 3D model provides the flexibility to locally vary the mining height to target narrower, higher-grade zones and to make adjustments for the vertical grade profile. This is especially true in localised areas proximal to the growth faults in Kansoko Sud, where the deposit was drilled at 50 m to 100 m grid spacing to account for the additional complexity. The 3D model is intended to provide the flexibility to make mining height or grade profile adjustments on a local scale to optimise the mine plan and improve Project economics.

A modified 2D modelling approach was used for the Kakula deposit where three grade shells were constructed to address the grade characteristics. A 3D model similar to the Kamoa model is currently in the planning stage for Kakula. Figure 14.34 through Figure 14.36 illustrate how the TCu grade increases and continuity of higher-grades improves using higher TCu cut-offs at Kakula.

Figure 14.34 Kakula: TCu Grades for the Overall 1.0% TCu Model

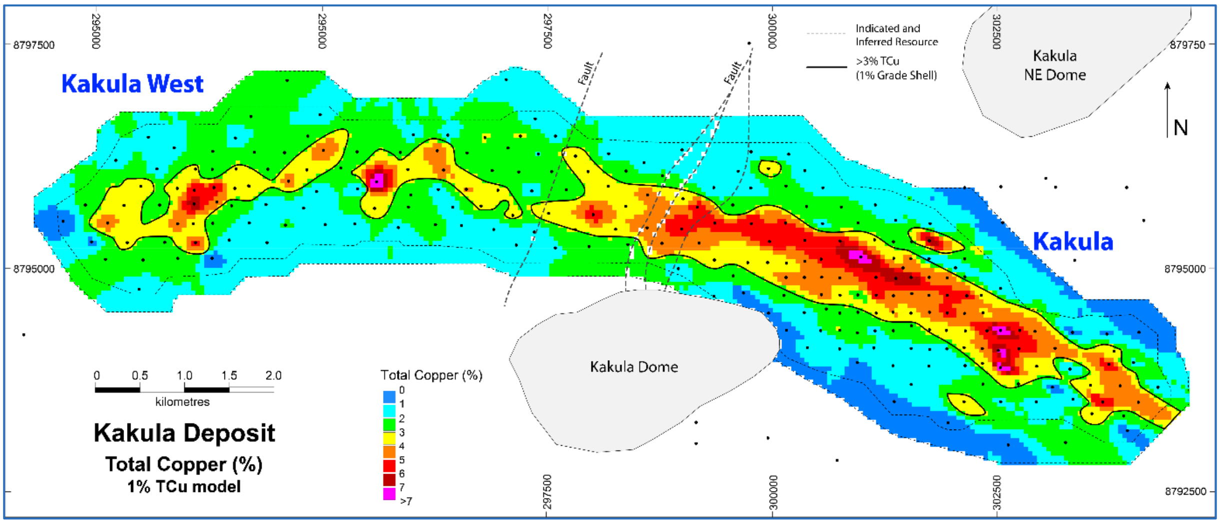


Figure provided by Ivanhoe, 2018. Model grades >3% TCu are outlined for the 1% TCu model (this overall model incorporates the SMZ10U, SMZ20U, SMZ30, SMZ20L and SMZ10L models). Missing blocks are due to non-vertical fault offsets.

Figure 14.35 Kakula: TCu Grades for the Overall 2.0% TCu Model

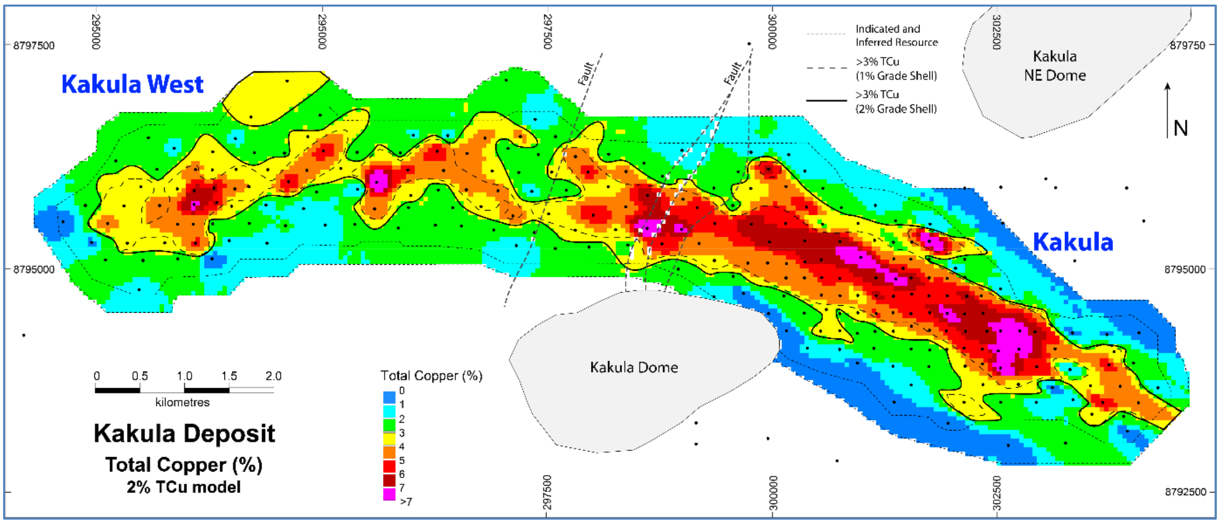


Figure prepared by Ivanhoe, 2018. Model grades >3% TCu are outlined for the 2% TCu model (this overall model incorporates the SMZ20U, SMZ30 and SMZ20L models). The original outline from the 1% model (stippled line) is shown to emphasise how the higher-grade area expands using higher TCu modelling cut-offs. Missing blocks are due to non-vertical fault offsets.

Figure 14.36 Kakula: TCu Grades for the 3.0% TCu Model

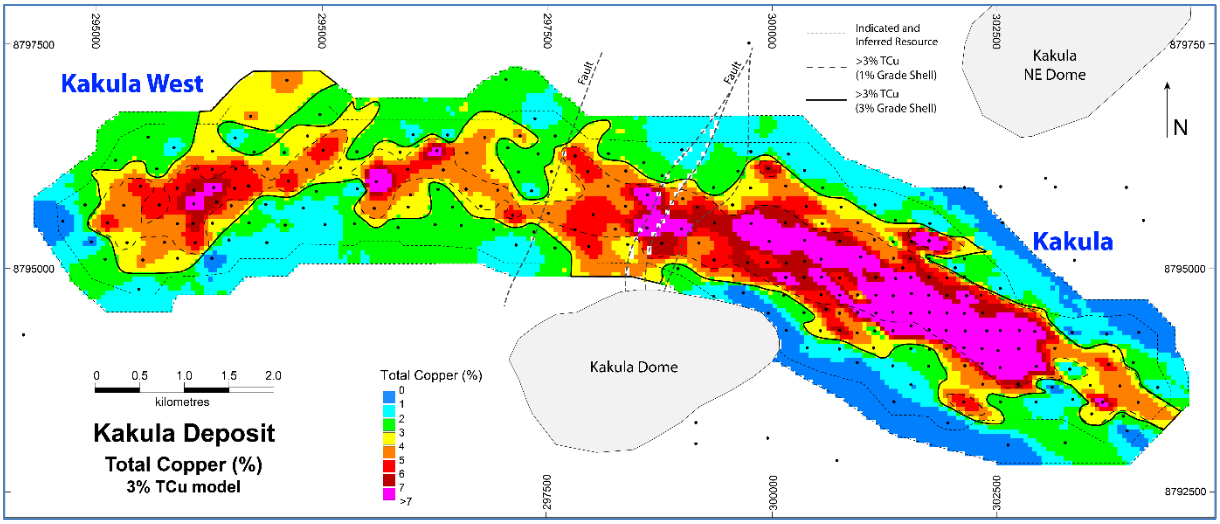


Figure prepared by Ivanhoe, 2018. Model grades >3% TCu are outlined for the 3% TCu model (the SMZ30 model). The original outline from the 1% model (stippled line) is shown to emphasise how the higher-grade area expands using higher TCu modelling cut-offs. Missing blocks are due to non-vertical fault offsets.

14.18 Targets for Further Exploration

Amec Foster Wheeler has identified a target for further exploration at Kamoa (referred to as an exploration target for the purposes of this Report). It is referred to in this subsection as the Kamoa–Makalu exploration target. No targets have been defined for the Kakula deposit as the limits of the mineralisation have not been established.

The area inside the model perimeter surrounding the Indicated and Inferred Mineral Resources is shown in Figure 14.37. The ranges of the Kamoa-Makalu exploration target tonnages and grades are summarised in Table 14.19. Tonnages and grades were estimated using SMZ10 composites in the target area and applying a +/-20% variance to the tonnages and grades.

Amec Foster Wheeler cautions that the potential quantity and grade of the Kamoa–Makalu exploration target is conceptual in nature, and that it is uncertain if additional drilling will result in the exploration target being delineated as a Mineral Resource.

Figure 14.37 Kamoa-Makalu Target for Further Exploration Location Plan

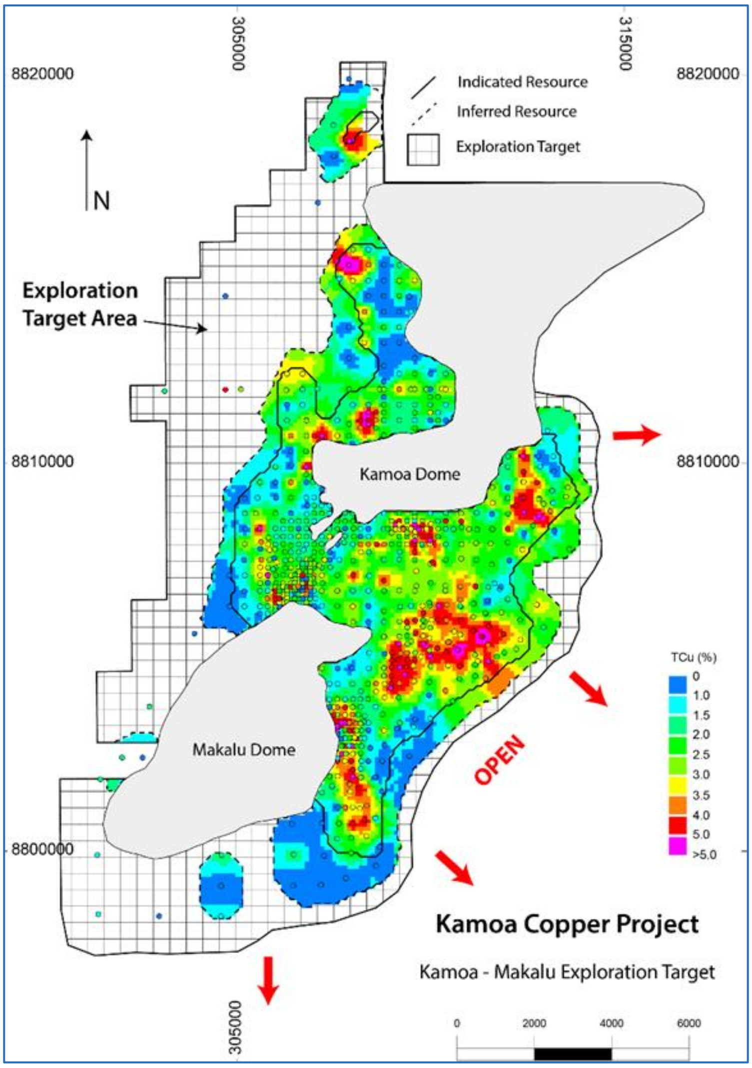


Figure by Ivanhoe, 2016. Scale bar represents metres.

Table 14.19 Kamoa-Makalu Target for Further Exploration: Tonnage and Grade Ranges

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

Additional Exploration Potential

The eastern boundary of the Mineral Resources at Kamoa is defined solely by the current limit of drilling, at depths ranging from 600 m to 1,560 m along a strike length of 10 km. Some of the best grade-widths of mineralisation occur here, and in addition, high-grade bornite dominant mineralisation is common. Beyond these drillholes the mineralisation and the deposit are untested and open to expansion, even beyond the Kamoa-Makalu target for further exploration.

At Kakula, the western and south-eastern boundaries of the high-grade trend within the Mineral Resources are defined solely by the current limit of drilling. Exploration drilling is ongoing in these areas, and there is excellent potential for discovery of additional mineralisation.

14.19 Comments on Section 14

Mineral Resources for the Project have been estimated using core drill data, have been performed using industry best practices (CIM, 2003), and conform to the requirements of CIM Definition Standards (2014). Amec Foster Wheeler has checked the data used to construct the resource model, the methodology used to construct them (Datamine macros) and has validated the resource model. Amec Foster Wheeler finds the Kamoa resource model to be suitable to support prefeasibility level mine planning, and the Kakula resource model is suitable to support a preliminary economic assessment.

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

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

  - Metallurgical testwork at the Kamoa deposit indicates the need for multiple grinding and flotation steps. Variability testwork has been conducted on only portions of the Kamoa deposit. Additional variability testing is needed to build models relating copper mineralogy to concentrate grade and improve the recovery modelling.
  - A basic model predicting copper recovery from certain supergene mineralisation types has been developed. More variability testing is required to improve this model to the point where it is useful for production planning purposes.
- Metallurgical recovery assumptions at Kakula.
  - Preliminary metallurgical testwork at the Kakula deposit indicates that a high-grade chalcocite-dominant concentrate could be produced at similar or higher recoveries compared to those achieved for Kamoa samples.
  - There is no supergene mineralisation currently identified at Kakula that requires a dedicated recovery model separate from the hypogene recovery prediction method.
- Exploitation of the Kamoa-Kakula Project requires building a greenfields project with attendant infrastructure. Changes in the assumptions as to operating and capital costs associated with the proposed development may affect the base case cut-off grades selected for the Kamoa and Kakula Mineral Resource estimates.
- Commodity prices and exchange rates.
- Cut-off grades.

15 MINERAL RESERVE ESTIMATES

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

The Kamoa 2017 PFS Mineral Reserve has been estimated by Qualified Person Jon Treen, Senior Vice President, Stantec Consulting LLC, using the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves to conform to the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects. The total Mineral Reserve for the Kamoa Project is shown in Table 15.1. The Mineral Reserve is based on the 2017 Mineral Resource. The Mineral Reserve is entirely a Probable Mineral Reserve that was converted from Indicated Mineral Resources. The effective date of the Mineral Reserve statement is 28 November 2017.

The Mineral Reserve defined in the Kamoa 2017 PFS has not used all the Mineral Resources available to be converted to Mineral Reserve, as the analysis was constrained to produce a production period of 26 years. The Mineral Reserve is entirely contained within the Kamoa Mineral Resource. The two main areas within the Kamoa Mineral resource are: the Kansoko Sud and Centrale areas.

Table 15.1 Kamoa 2017 PFS Mineral Reserve Statement

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

1. Effective date of the Mineral Reserve is 28 November 2017.
2. The copper price used for calculating the financial analysis is long-term copper at US$3.00/lb. The analysis has been calculated with assumptions for smelter refining and treatment charges, deductions and payment terms, concentrate transport, metallurgical recoveries and royalties.
3. For mine planning, the copper price used to calculate block model Net Smelter Returns was US$3.00/lb.
4. An elevated cut-off of US$100.00/t NSR was used to define the stoping blocks. A cut-off of US$80.00/t NSR was used to define ore and waste for the mine plan.
5. Indicated Mineral Resources were used to report Probable Mineral Reserves.
6. The Mineral Reserves reported above are not additive to the Mineral Resources.

The production plan defined for the Kamoa 2017 PFS Mineral Reserve represents the first phase of the strategy defined by the Kamoa-Kakula 2017 PEA.

The Kamoa 2017 PFS Mineral Reserve ranges between depths of 60 m and 1,300 m below surface, and the average dip is approximately 17°. Given the favourable mining characteristics of the Kamoa Mineral Resource, it is considered amenable to large-scale, mechanised, room-and-pillar mining or controlled convergence room-and-pillar mining. The saleable product will be copper concentrate. The processing production rate is 6 Mtpa ore.

The room-and-pillar mining will be used for ore zones from 60 m to 150 m in depth and from 150 m to 250 m in depth selectively during the production ramp-up period. For ore zones below 150 m not mined room-and-pillar during the ramp-up, controlled convergence room-and-pillar is the mining method of choice. Dip, depth, and mining height will define pillar size and post-destructive recovery. No postmining backfill will be required with these two methods.

Dilution has been applied as waste skins at the top and bottom contacts and by the use of footwall wedges below the orebody. Dilution was determined based on the method and shape, and mining losses were estimated as 2% for development and 5% for pillar extraction, to account for unrecovered ore.

Separate recoveries were applied to the Supergene and Hypogene metallurgical ore types. Smelter terms, concentrate transport, and royalties were applied to calculate the block model NSR. The NSR used for the Mineral Reserve definition assumed that concentrate transport was by road.

An NSR cut-off of $100.00/t was used to define the stoping blocks. An NSR cut-off of US$80.00/t was used to define ore and waste for the mine plan. Both these cut-offs are elevated relative to the breakeven cut-off that can be calculated from the cost assumptions in the economic analysis of the Kamoa 2017 PFS. The process, G&A, and mining costs that equate to the breakeven cut-off grade are approximately $46/t ore (see Table 21.2).

The Mineral Reserve will be impacted by changes in revenue, costs, and other parameters. The elevated cut-off grades used to define the Mineral Reserve are a buffer against increases in cost or reduction in grade or recovery. The methodology used to define the Mineral Reserve has resulted in the highest-grade mining zones being identified to be mined first; this means that if the parameters vary positively or negatively, then it is likely that the mine plan, including the order of mining, will not change significantly.

As the mining production period was arbitrarily defined as 26 years, it is likely that further studies will define additional Mineral Reserves. This is supported by the large Mineral Resource that has already been defined.

Power supply to the project and continuity of supply are important factors that can affect the Mineral Reserve. To reduce the risk to the project, capital has been included for the power station upgrade to secure power for the project. This also allows more detailed studies to be undertaken to optimise the Kamoa production capacity.

In the economic analysis, it has been assumed that rail will be available after two years and that there is therefore a significant reduction in concentrate transport costs, relative to the road transport assumption. This also provides a buffer against a reduction in Mineral Reserve.

16 MINING METHODS

16.1 Geotechnical

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

This section contains a summary of the PFS-level mining geotechnical investigation and design conducted for the Kamoa 2017 PFS and shown in Section 16.1.1.1 and the PEA-level geotechnical investigation carried out for the Kamoa-Kakula 2017 PEA and shown in Section 16.1.2 .

16.1.1 Kansoko Geotechnical Investigation and Design

The geotechnical investigation was based on geotechnical drilling and logging conducted by Ivanhoe over the Kamoa project area, which was reviewed and interpreted by SRK and is essentially unchanged from the previous studies (Kamoa 2013 PEA, Kamoa 2016 PFS and Kakula 2016 PEA) in order to provide geotechnical designs for the room-and-pillar method incorporated in the Kamoa 2017 PFS. Geotechnical design for the room-and-pillar method was also carried out by SRK and is also unchanged from the Kamoa 2016 PFS. The controlled convergence room-and-pillar method incorporated in the Kamoa 2017 PFS was developed and designed by Cuprum (2016, 2017a, and 2017b) and reviewed by SRK for the purposes of this report.

16.1.1.1 Geotechnical Database

Geotechnical drilling and logging was conducted by Ivanhoe. SRK completed three site visits to the Kamoa-Kakula Project during 2011 for the purposes of geotechnical and structural logging QA/QC and data quality control.

Findings from the visits have been documented in two memoranda (Jakubec, J. 2010, 2013) which provide outline on-site protocols, quality control reviews, details of the findings, recommendations for future data collection, and update aspects of various geotechnical and mining studies. Limited on-site data analysis and preliminary findings are also documented. Recommendations have been made for regular follow-up visits as the project study level, data quantity, and required level of detail increases.

The geotechnical data collection is acceptable at this stage of the project, but there are a number of areas that require improvement as the project continues:

- Geotechnical data collection: Geotechnical parameter collection is considered to be fair, with ongoing issues noted relating to RQD measurements (inclusion of mechanical breaks). However, the identification of natural versus mechanical breaks is being completed to a high standard. Intact rock strength is locally underestimated; however, in most cases the patterns of strength change are being identified.
- Orientation data collection: Alpha orientation measurements (angle of the break to the core axis) are being collected to a very high standard. Conversely beta measurements (angle of the maximum dip of the fracture related to the reference line) are being collected poorly with errors noted in identification of maximum dip vector, downhole direction, and actual measurement.
- Geotechnical database: The Kamoa geotechnical database was considered to be of fair quality during the audit. While some inherent issues existed, the process of filtering and cleaning the dataset will improve the quality of the geotechnical dataset. SRK understand that significant work has been undertaken recently to improve this.
- Geotechnical recommendations: Several changes have been made to structural and geotechnical data collection processes recently based on the recommendations by SRK in August 2010 and June 2011. Time should be taken to make sure that these changes are carried out correctly during the early stages of implementation. Additional quality control checks by Kamoa Copper SA geotechnical engineers have been recommended at all stages of data collection.

The status of the structural data being collected has been reviewed. It was decided that the current fault network interpretation cannot be further developed with current information. More detailed structural logging has been recommended and the data capture is underway. Once a more complete set of structural logs are available for the available drill core, further interpretation should be undertaken to improve the structural/geotechnical domains.

16.1.1.2 Geotechnical and Structural Models

Structural Domains

The 2012 structural model was updated in 2013/2014 to include the new drilling data and define a primary fault network for the geotechnical studies that could also be used for updating the resource model. The model also needed to be updated in order to reach as near a PFS level of structural understanding as possible, given the scale of the project and lack of outcrop exposure.

During this study, the previously identified faults have been placed into a more robust tectonic framework. The understanding of the age and nature of structural development within the study area has been changed and improved. The new model consists of 45 faults divided into six dominant sets of differing orientations. To assist with the interpretation, other data sources including topographic analysis and surface geophysics were used.

Structural domains together with rock mass data collected are presented in Figure 16.1.

Joint Patterns

There are 3 main joint sets present across the project area:

- A steep north-north-east joint set;
- A shallow dipping set parallel to bedding dip; and
- A generally steep east-west trend.

The north-north-east striking sub-vertical and shallow bedding plane joint sets are pervasive throughout the area. The east-west joint trend is limited to areas labelled 3A, B, and C in Figure 16.2.

The joint patterns will have a bearing on the anticipated hangingwall deformation and support requirements, pillar strengths and performance characteristics. Therefore, cognisance of the joint patterns is essential during the mining method design. This has been taken into account with the CUPRUM mine method design.

Figure 16.1 Structural Domains and Rock Mass Values

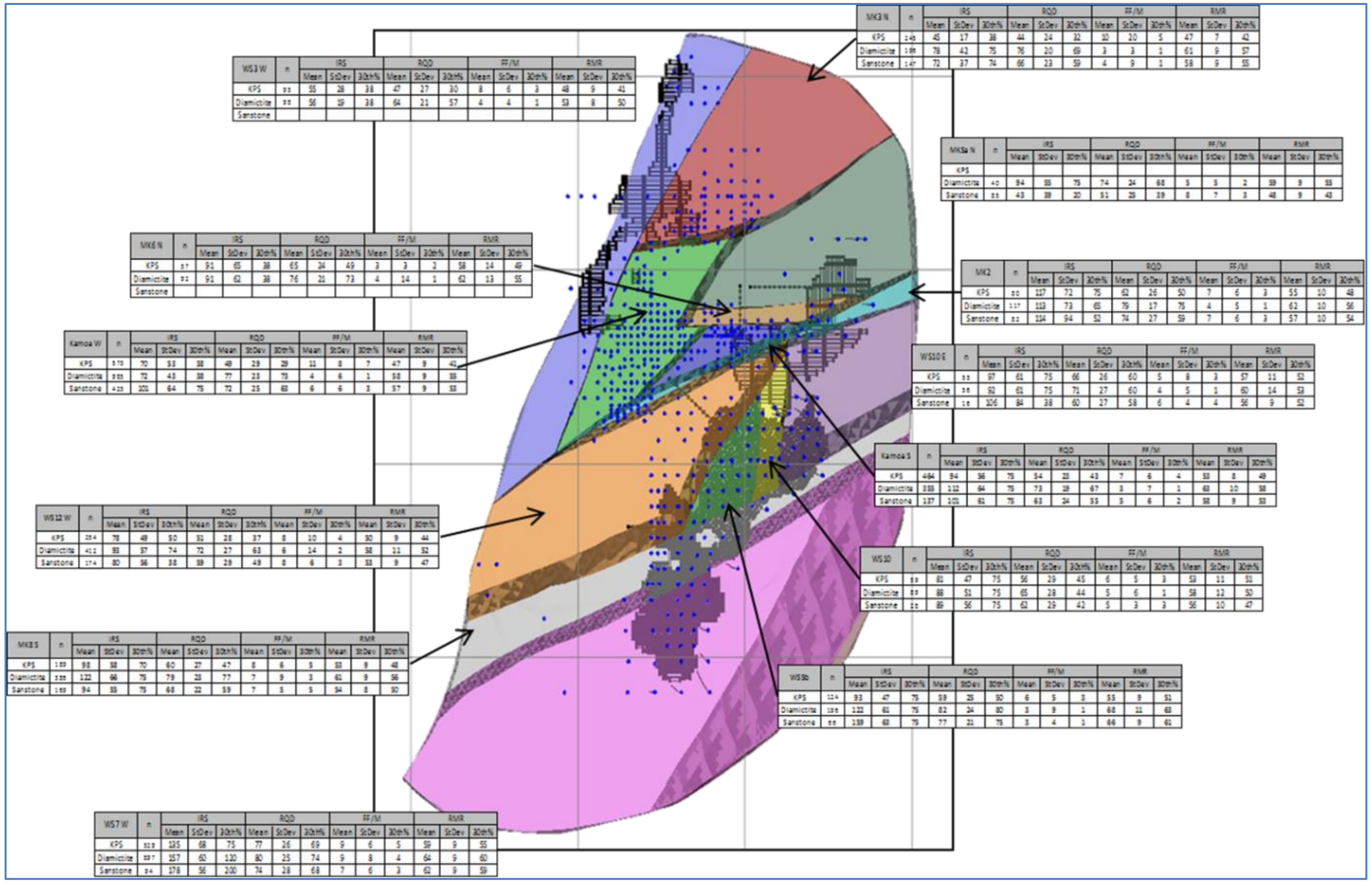


Figure 16.2 Overview of Joint Pattern Variations Across the Kamoa Project, Mapped Outline in Background

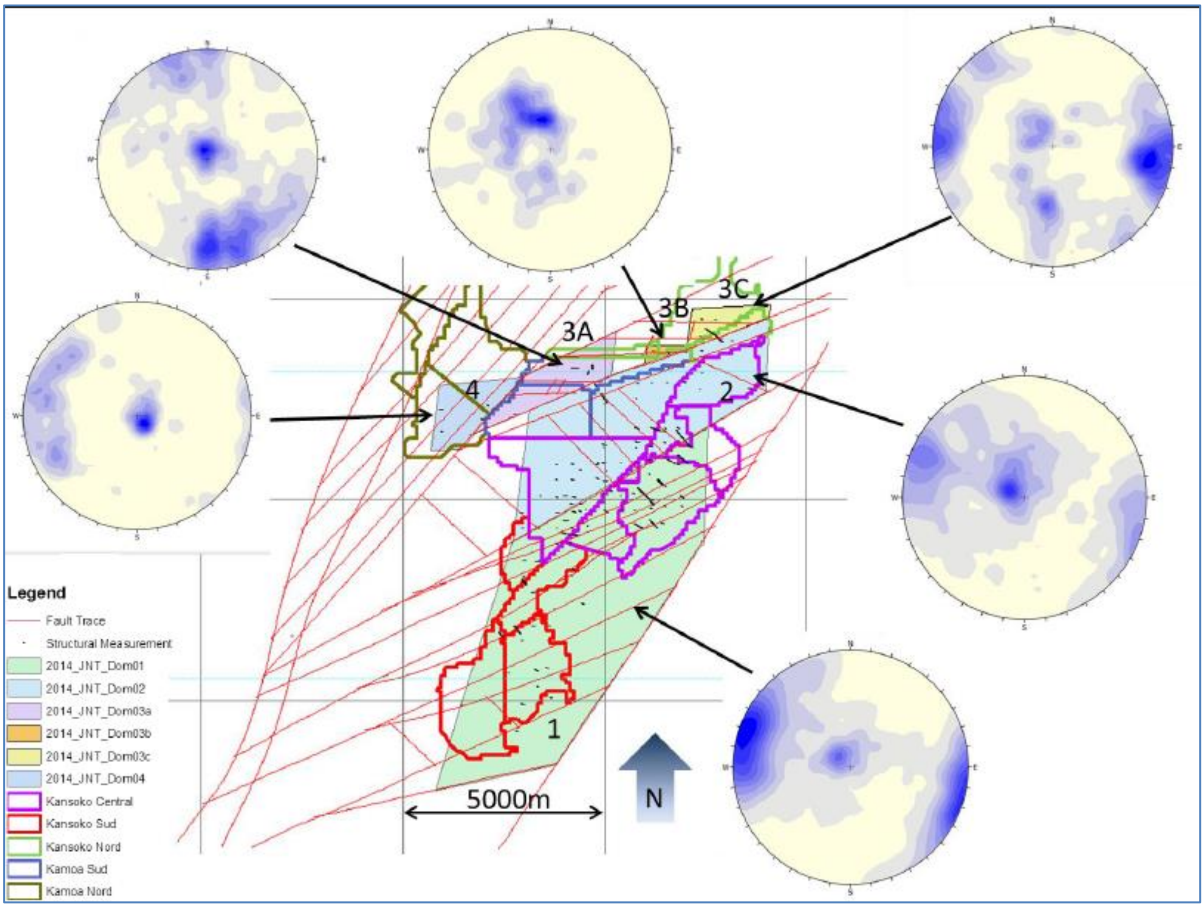


Figure by SRK, 2017.

Weathering

Weathering of the rock mass is highly variable at Kamoa. Structurally controlled weathering appears to extend to considerable depth (over 500 m below surface) in places.

The KPS siltstone is a stratigraphic layer above the orebody, which has a tendency to weather rapidly and the distance between the roof and this layer varies considerably and in some areas, it forms the hangingwall to the deposit. A safe distance between the roof of the mining operation and the KPS should be maintained to a minimum of 3 m.

Geotechnical Domains

The assessment of the geotechnical properties assumes three primary lithological domains, namely siltstone, diamictite and sandstone (from the Roan basement rock). The orebody is located primarily within the diamictite. An additional geotechnical domain can be defined that consists of the weathered rock mass at surface. SRK modelled the base of the weathered rock based on the weathering descriptions in the provided drill logs.

A statistical approach was used to evaluate the data (separated by lithology), resulting in primary geotechnical division of the rock mass based on weathering. The weathering category was used to establish fresh, moderately weathered, and extremely weathered geotechnical domains. 3D wireframes were developed with average thickness of 10 m (extremely weathered) and 45 m (moderately weathered).

The established structural domains were used to further subdivide the data, with four fresh rock geotechnical domains established. The near surface Extremely Weathered Geotechnical Domain was not considered further for the underground geotechnical study.

The geotechnical parameters for intact rock strength, RQD, fracture frequency, joint condition rating, and RMR 89 for each geotechnical domain are presented in Table 16.1. Figure 16.3. shows a plan view of three fresh geotechnical domains: north; central and south.

Figure 16.3 Plan View of Three Fresh Geotechnical Domains (North, Central, South)

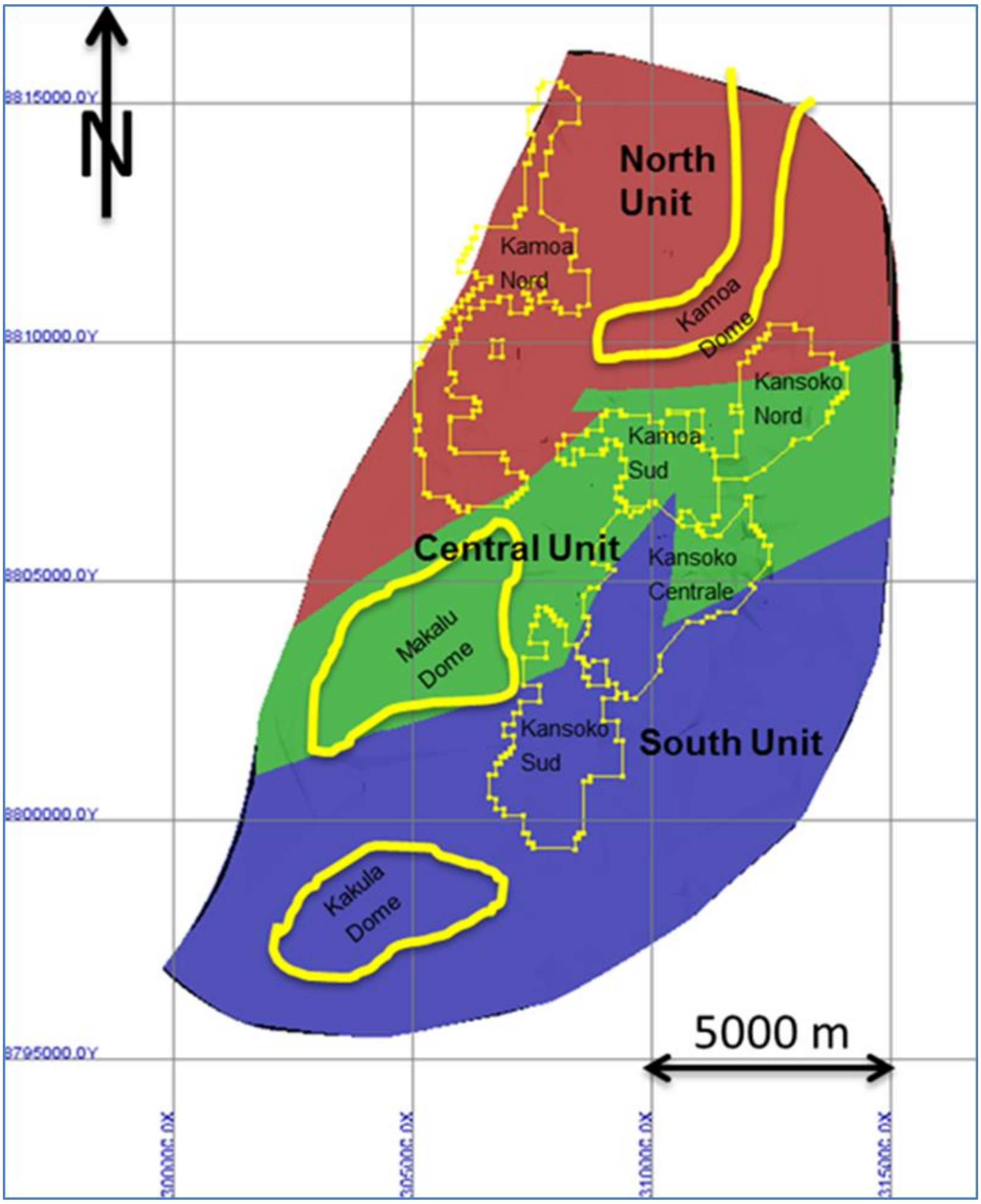


Figure by SRK, 2017.

Table 16.1 Summary of Geotechnical Parameters per Geotechnical Domain

Domain	Stratigraphy	RQD (%)	FF/m	RMR_{89}	Intact Young's Modulus (GPa)	Poisson's Ratio
Moderately Weathered	KPS	40	10	43	35 Est	0.24 Est
	Diamictite	63	6	51	47	0.28
	Sandstone	55	8	48	32	0.22
Fresh, North	KPS	48	11	47	66	0.28
	Diamictite	76	4	58	67	0.27
	Sandstone	67	6	56	58	0.23
Fresh, Central	KPS	55	7	53	66	0.28
	Diamictite	73	5	60	67	0.27
	Sandstone	62	6	55	58	0.23
Fresh, South	KPS	68	8	56	66	0.28
	Diamictite	80	8	63	67	0.27
	Sandstone	71	6	59	58	0.23

Rock Properties

Geomechanical laboratory testing was undertaken during 2012 and 2013 by SRK Canada a total of 121 samples were tested to determine uniaxial compressive strength (UCS). The engineered intact rock strength (IRS) presented in Table 16.7 considers the field estimated IRS (as logged by African Mining Consultants), field point load testing, and laboratory unconfined compressive strength testing. Table 16.7 lists the mean values, standard deviation (in brackets), and the derived engineered intact UCS of each stratigraphy within geotechnical domains. Due to a lack of data coverage across the deposit, the UCS data has been repeated in each domain for comparison to other data sources.

Table 16.2 Summary of Intact Rock Strength Estimates per Geotechnical Domain (Standard Deviation in Parenthesis)

Domain	Stratigraphy	Logged IRS (MPa)	# of UCS Tests	Intact UCS (MPa)	Point Load Test (MPa)		Engineered IRS (MPa)
					Axial	Diametral	
Weathered	KPS	45 (32)	1	123	75 (51)	61 (42)	45
	Diamictite	44 (30)	8	56 (31)	66 (54)	50 (30)	50
	Sandstone	44 (31)	4	153 (48)	88 (56)	81 (52)	75
Fresh, North	KPS	63 (47)	8	208 (36)	67 (43)	66 (44)	90
	Diamictite	72 (42)	17	98 (29)	97 (55)	71 (35)	100
	Sandstone	86 (59)	2	219 (22)	96 (55)	86 (59)	100
Fresh, Central	KPS	91 (57)	8	208 (36)	115 (54)	108 (54)	90
	Diamictite	101 (62)	17	98 (29)	80 (44)	92 (43)	100
	Sandstone	91 (63)	2	219 (22)	132 (50)	112 (60)	125
Fresh, South	KPS	116 (49)	8	208 (36)	144 (66)	140 (56)	120
	Diamictite	143 (64)	17	98 (29)	108 (48)	106 (39)	125
	Sandstone	131 (69)	2	219 (22)	121 (60)	126 (49)	125

Furthermore, a thorough laboratory testing programme was undertaken by Cuprum in 2015 to establish an extended stress – strain correlation in the post failure phase of the diamictite (mineralised zone). The laboratory testing established the average post failure strength for siltstone was approximately 14% of the UCS for siltstone and 16.5% for diamictite. It is noted that these stress – strain correlations correspond well with the dolomite rock mass being extracted according to Cuprum (2017a) at a number of copper mines in Poland using the controlled convergence room-and-pillar mining method.

16.1.1.3 Geotechnical Design

The mining methods and layouts are described in Section 16.2. The two mining methods to be used are room-and-pillar and controlled convergence room-and-pillar.

The room-and-pillar design provides for a stiff, non-yielding system in which excavations remain open for the LOM and primary infrastructure and access ways (declines and strike drives) are accessible without interruptions, all the way from the mining front back towards the centre of the mine. However, this mining method significantly reduces the extraction ratios.

Subsequently, a strategic decision to change the mining method was made by Ivanhoe, after a visit to KGHM Polska Miedź S.A in Poland to view a controlled convergence room-and-pillar mining method used by their Polish copper mines. This mining method is used by a number of copper mines in Poland, which therefore provide a basis for comparison with regard to attributes and efficacy of the method. This approach provides for a "controlled" goafing of the back area under the action of smooth (continuous) hangingwall closure, and rests on the principle of crushed pillars providing a residual support capacity directly after being cut at the advancing face.

Geotechnical Discussion of Mining Methods

Room-and-pillar will be used up to a depth of 150 m, to limit the risk of subsidence. There is abundant experience in the application of room-and-pillar mining to tabular orebodies in a wide range of geological environments. It is notable that large-scale room-and-pillar mining have been associated with unexpected massive collapses due to sudden failures over an extensive area. The key requirements for successful application of the room-and-pillar method is a proper understanding of the stability of the rooms and ensuring that the in-panel pillar layouts are adequate for the expected conditions. Taking this into account the room-and-pillar mining method has been adequately designed.

The controlled convergence room-and-pillar mining method has an aggressive in-panel recovery layout where pillars are designed on experience to yield progressively as the mining advances. The planned application of the controlled convergence room-and-pillar method rests on the premise that the same method has been successful at mines in Poland owned by KGHM Polska Miedz SA (KGHM) (Cuprum, 2017a). This premise has been thoroughly assessed by Cuprum (2015) where the following geotechnical parameters for the hangingwall, orebody and footwall have been compared:

- Geology (orogeny, stratigraphy).
- Rock mass strength and performance characteristics (laboratory tests, rock types, rock mass classification, local tectonic disturbances).

It should be noted that the risk to the roof stability that exists in the KGHM mines also exists in the Kansoko mining area. The problems encountered and strategies applied to mitigate against these problems at KGHM's Polish mines have been recommended for the Kansoko Mine.

The layout is geometrically well-defined with dip and strike barrier pillars and panels up to 300 m wide and 500 m long.

Overall, the controlled convergence room-and-pillar methodology appears to be suitable for the Kansoko mining area, however, a focussed assessment of the hangingwall conditions must be carried out in the next phase of study. In addition, the underground development and trial stoping is required to better understand the geological structure and its potential influence on the mine design.

Pillar Design (Room-and-Pillar)

Pillar strength for the room-and-pillar mining method has been designed adequately to provide for a stiff, non-yielding system for the areas where this mining method is to be used.

The pillar design and theoretical extraction ratios for a range of depth intervals and mining heights were based on Tributary Area Theory (TAT) for square and rectangular pillars. The Kamoa resource and surrounding rock conditions change significantly across the project area. A variety of pillar designs are provided to accommodate the changing rock mass conditions.

Geotechnical logging and laboratory tests were used to derive a Design Rock Mass Strength (DRMS) equivalent to the strength of 1 m^3 in the pillar. Laubscher's 1990 method was used to determine the DRMS.

In-panel pillar designs are based on the Hedley and Grant (1972) empirical formula. The formula derives the in-situ pillar strength from the DRMS and the pillar dimensions. The strength of the stability pillars was assessed based on the empirical relationship after Stacey and Page, considering a panel length (500 m) and a width of 40 m.

The in-panel pillar loads were initially calculated using the Tributary Area Theory (TAT) that assumes that pillars carry the entire load to surface and this is shared equally by all the pillars. The pillar load is a function of the virgin vertical stress and extraction ratio.

The extraction ratios calculated by CUPRUM for the room-and-pillar mining area are shown in Table 16.3.

Table 16.3 Room-and-Pillar Mining Method Extraction Ratios

Dip Intervals (degrees)	Mining Height Intervals in Panel (m)	Extraction Ratio (60–150 m) (%)	Extraction Ratio (150–250 m) (%)
0–12	3 to ≤4	79	75
13–16		73	71
17–20		74	71
21–25		72	72
26–30		71	69
31–35		69	66
0–12	<4 to ≤5	75	72
13–16		72	69
17–20		72	70
21–25		66	64
26–30		64	62
31–35		63	60
0–12	<5 to ≤6	72	69
13–16		70	67
17–20		70	68
21–25		64	62
26–30		63	61
31–35		62	59

Pillar Design (Controlled Convergence Room-and-Pillar)

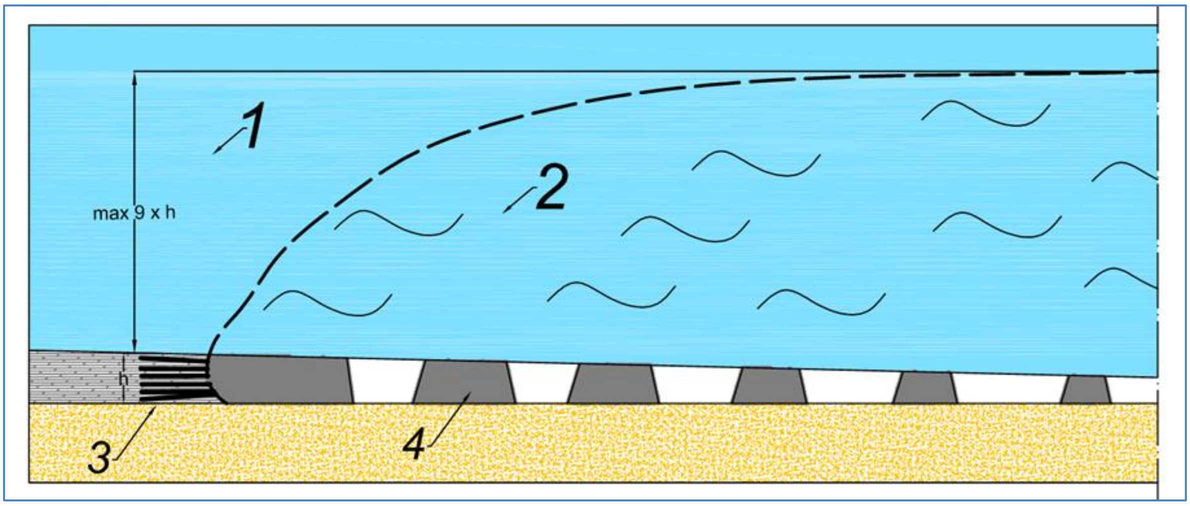
Pillar strength for the controlled convergence room-and-pillar mining method is premised on a percentage of the UCS for the post-failure strength estimate. The results of 14% and 16.5% of the UCS are reasonable quantities (~20 MPa) post failure strength. This is the strength assigned to the pillars (or rather, the "pillar cores") for the pillar design. The extraction ratio for the controlled convergence room-and-pillar mining method are shown in Table 16.4.

The anticipated mode of failure/deformation of the hangingwall appears to be a controlled closure of a continuous stratigraphic horizon as the back-area pillars deform in post failure mode as depicted in Figure 16.4. The Cuprum (2016) report does not specifically state that it takes into account the potential influence of structural discontinuities that may result in wedge or structural failure or how the outreach of the distressed rock mass area will enable smooth roof bending strata. However, these scenarios are recognised and provision made in the form of recommendations that hydraulic props, wooden cribs and cable bolts must be used in areas where complex geological or difficult mining conditions exist.

Table 16.4 Controlled Convergence Room-and-Pillar Mining Method Extraction Ratios

Dip Intervals (degrees)	Mining Height Intervals in Panel (m)	Extraction Ratio (Primary Phase = Face Blasting Works) (%)	Extraction Ratio Secondary (From Pillars Scraping) (%)	Total In-Panel Extraction (%)
0–12	3 to ≤4	66	24	90
13–16		65	24	89
17–20		64	25	89
21–25		62	27	89
26–30		52	33	85
31–35		49	36	85
0–12	<4 to ≤5	62	27	89
13–16		60	28	88
17–20		59	29	88
21–25		56	31	87
26–30		46	38	84
31–35		44	40	84
0–12	<5 to ≤6	56	31	87
13–16		55	31	86
17–20		55	32	87
21–25		53	33	86
26–30		44	39	83
31–35		42	41	83

Figure 16.4 Controlled Convergence Room-and-Pillar Rock Mass Impact



1 – rock mass prior to extraction; 2 – destressed and delaminated rock mass; 3 - blasting holes; 4 - primary pillars. Figure by KGHM Cuprum, 2014.

Protection of Main Access Ways

Cuprum (2016) has stipulated that the declines and underground chambers will be protected using 20 m protection pillars on either side of the decline array and underground chambers where the workings are shallower than 600 m. The width of this pillar will increase to 40 m for depths below 600 m. Figure 16.5 and Figure 16.6 show the protection pillars for declines of mining dip <12 degrees and 13 to 16 degrees, respectively.

The scenarios in both Figure 16.5 and Figure 16.6 show that mining progresses towards the declines and this is in effect, a retreat mining methodology. The secondary development is only required ahead of the mining faces and should not be required in the back areas where the controlled convergence room-and-pillar has occurred. There is however a concern when reviewing the mine design/mining direction that the mining direction is away from the main access decline. In this situation additional pillars may be required on the goafing side of the secondary access ways to protect these access ways for the life of the panel specifically to contain any unravelling, wedge failure which may result in closure of these access ways. This has been taken into account Cuprum (2017b) report where it is stated that a protection zone will be required for the secondary drives that are required for access to other mining panels. Provided that the protection zone (including that applicable to secondary drives) is implemented, then it can be concluded that the access ways are adequately protected.

Figure 16.5 Controlled Convergence Room-and-Pillar Mining Method and Pillar Geometry for a Deposit Dip up to 12 Degrees

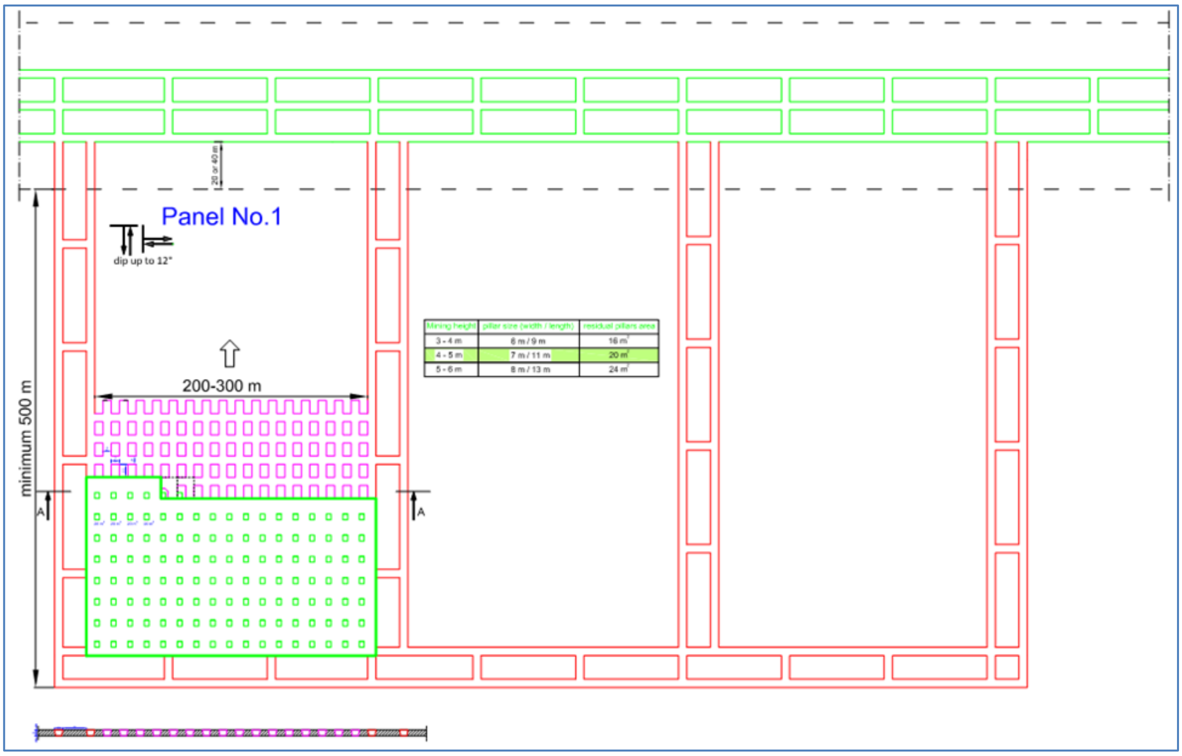


Figure by KGHM Cuprum, 2017

Figure 16.6 Controlled Convergence Room-and-Pillar Mining Method and Pillar Geometry for a Deposit with Dip Angle of 13 to 16 Degrees

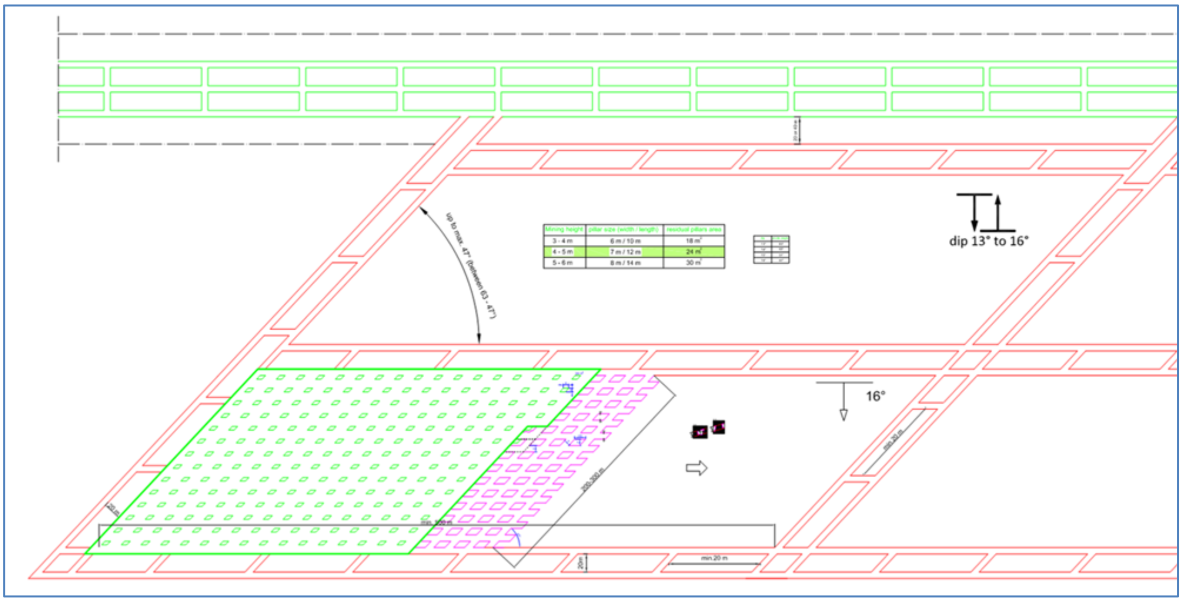


Figure by KGHM Cuprum, 2017

Numerical Analysis

The two-dimensional software program Phase 2 (version 8) was used by Cuprum to conduct numerical modelling to determine the minimum width of the protection/safety pillars. An elastic, perfectly plastic material for the roof, orebody and the footwall were used. The mining in the vicinity of a long-term excavation (main access way, etc.) were included in an attempt to establish both the size of the required protection/barrier pillar and the overall stability of the excavation. The input parameters were appropriately determined using the laboratory test results and Rocklab software.

The modelling exercise and software code is suitable to determine the safety pillar dimensions and the height of the tensile zone of the long-term excavation at this level of study.

It must be borne in mind that that all models are a simplification of reality and consequently, the residual load applied to the barrier pillars is not completely understood. It is important to estimate or anticipate the potential for pillar punching into the footwall, seismicity (bursting of the pillars), shear failure of the hangingwall around the pillars and minimum pillar sizes required for the barrier (protection) pillars. Similarly, the height effect of irregular geometries (i.e. where one side is significantly taller than another) needs to be investigated in the next phase of study.

The actual response of the hangingwall, footwall and in-panel pillars needs to be quantified during a mining trial. Detailed monitoring as described by Cuprum (2016) will be essential to obtain the maximum benefit of the trial. This data then needs to be fed into a 3D in-elastic numerical model such as FLAC to obtain a better understanding of the rock mass behaviour associated with this mining method.

Support Design

Support design uses the principle of the height of a de-stressed "tensile" zone within the hangingwall above the exposed room and a suitable factor of safety (FoS). Cuprum (2016) typically recommend tendons in the order of 2.0 m and spaced apart at acceptable intervals. At this stage of the study, the review finds that the support recommendations appear to be reasonable but can be adjusted as necessary going forward.

However, where the back-area pillars will continue to lose strength and hangingwall closure will increase, it is important to understand if the hangingwall will behave as a continuous stratigraphic horizon. If so, tendons of length, spacing and capacity are adequately designed as a function of the tensile zone above the excavated rooms.

Alternatively, if the hangingwall does not behave as a continuous stratigraphic horizon but is expected to unravel or undergo wedge failure or parting on a stratigraphic contact, wide-scale falls of ground (FoGs) may occur in the back area. Such FoGs may either reduce the extraction ratio and/or result in catastrophic closure of critical accesses if the failure is not managed proactively. It is understood that the project will be carried out under a comprehensive monitoring programme, as a trial, and the mining direction, additional support and monitoring systems appear to have been adapted to anticipate and manage this mode of failure.

Preliminary Subsidence Review

Surface deformations will begin to reveal themselves in the first five years after mining begins in the area located to the south of the mine buildings. At first, two local troughs will develop, and the maximum surface subsidence will not exceed 0.4 m. The impact of drainage on surface deformation will be negligible during the first five years. By analysing the picture of total surface subsidence in the target period, it can be stated that a large field panel of displacement with two local centres will be created at the mine site of the designed mine, which will form themselves over both mine regions (Centrale and Sud). The larger trough will develop in the area of the Centrale region and its maximum subsidence will be about 2.7 m. What is important, this trough will be steep but strongly restricted spatially from the west side of the main plant buildings.

The displacement connected with of the south area (Sud region) should not exceed a maximum of 2.6 m. From the north, this local displacement field panel will exhibit significant slope of the trough profile coming up to 10.0 mm/m, accompanied by horizontal deformations with the maximum values of ±5.5 mm/m.

Water Management

This is discussed in the Cuprum (2016) report however the management of water is not specifically addressed. The goafing of the hangingwall will significantly increase permeability of the overlying rock mass. Golder's groundwater model takes cognisance of a goafing zone, equal to 2 x mining height and a damage zone, equal to 9 x mining height. They report an increase of inflow due to goafing of 12% to 30% in different modelled scenarios. It is important that this is taken into consideration in the design.

16.1.1.4 Conclusions and Potential Geotechnical Risks

The geotechnical data has been collected according to internationally acceptable standards and QA/QC reviews were done onsite to confirm compliance to data collection standards. Rock material testing of the main lithological has been done to establish typical rock mass strengths and elastic properties. Overall the work done is suitable for PFS requirements.

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

- The uncertainty due to the wide-spacing of data and lack of understanding of the frequency of structures and their deformation zones, that may impact the competency of the underground rock mass and the continuity of the deposit.
- The actual response of the hangingwall, footwall and in-panel pillars needs to be quantified during a mining trial. Detailed monitoring as described by Cuprum (2017a) will be essential to obtain the maximum benefit of the trial.
- The stress environment is unknown at this stage and it appears that Cuprum has used a k ratio of 1. However, if the horizontal stress is significantly high this will could result in an increase in the depth of failure in the hangingwall of the long-term excavations.
- The pillar design and extraction percentages are based on the summarised data obtained from drillhole core only. The information is considered representative but needs to be verified through data collection from underground exposures.
- Good quality conventional blasting appears to have been assumed in Cuprum’s analyses (2016, 2017a, and 2017b), where limited overbreak occurs. Poor blasting results in smaller- and taller- than designed pillars, negatively impacting on the pillar and span stability. Failure to achieve good quality blasting will significantly affect pillar performance.
- In portions of the various mining areas, the KPS lithology has been interpreted to possibly form the hangingwall package, and in some situations the upper portions of the pillars. Fresh KPS is not considered to be a concern for the rock mass, but exposed and weathered pyritic siltstone could rapidly degrade and cause significant hangingwall stability problems.
- The stability of the rock mass within the mining environment is not well known, specifically with respect to geological structures contained in the pillars and the hangingwall.
- It is important to retreat towards a stable access during pillar extraction to ensure safety of personnel.

16.1.1.5 Recommendations for Feasibility Study

The geology in the area includes significant geological structure with numerous faults and a wide range of joint orientations. The occurrence and the condition of these structures needs to be better understood. The behaviour of the hangingwall will be affected by geological structure and orientation. It is recommended that a full scale geotechnical mapping of the rock mass is done during the development and trial mining phase.

The main requirements for the successful implementation of the controlled convergence room-and-pillar mining method includes a good understanding of the hangingwall stability of the rooms during primary and secondary stages as well as confidence that the post peak performance of crush pillars are adequate for the expected conditions. The in-panel recovery layout is aggressive and although it appears to be suitable for the Kansoko deposit concerns have been raised by Cuprum (Cuprum 2017a, page 92) about the potential for a cave in hazard. Cuprum however, has carried out a thorough comparison of the geological and geotechnical parameters for the KHGM mines and concluded that the Kansoko rock mass was similar to that in Poland. SRK's review of this work indicates that this is a reasonable assumption and suitable for a PFS.

The stability/barrier pillars must be large enough to prevent undue damage to the long-term access ways and major infrastructure. The numerical modelling done is suitable for this level of study and indicates that the required size of the stability pillars fulfils the requirements of a PFS.

A proper understanding of the pillar strength and post failure strength of the pillar is critical to the project to ensure both stability and facilitate the required extraction.

The actual response of the hangingwall, footwall and in-panel pillars needs to be quantified during a mining trial. Detailed structural and geotechnical mapping and geotechnical instrumentation programme in the proposed trial site as described by Cuprum will be essential to obtain the maximum benefit of the trial.

The expected deformability of the pillars, the stability of the barrier pillars, strengths and expectation of the main access ways to remain open for the life of the panel needs to be quantified during the next stage of the study using data obtained during the trial mining process.

The stress environment is unknown at this stage and it appears that Cuprum used a k ratio of 1. However, if the horizontal stress is significantly high this could result in an increase in the depth of failure in the hangingwall of the long-term excavations. It is recommended that initially the modelling is done with varying k ratios to determine the potential effect of high horizontal stress. In addition to this it is recommended that during the trial period stress measurement is done to establish the magnitude and direction of the virgin stress.

The layout and sequences may need to be optimised in the next phase of study to ensure safety of personnel.

16.1.2 Kakula Geotechnical Investigation

This section presents the findings of the geotechnical investigation conducted by SRK Consulting in aid of the exploration programme at Kakula. The principal objectives of the investigation was to assess the ground conditions across the Kakula project area and delineate ground control districts (GCDs) based on data which is of sufficient quality and quantity.

16.1.2.1 Geotechnical Model

The geotechnical investigation is based on an assessment of all the available geotechnical data, laboratory test data and the projects geological and structural setting.

Laboratory rock strength testing which comprised Base Friction Angle (BFA) tests, Indirect Brazilian tests (BTS), Triaxial Compressive Strength (TCS) tests and Uniaxial Compressive Strength (UCMS) tests, including Young's Modulus and Poisson's Ratio determination were conducted for the following purposes:

- Rock mass characterisation.
- For the understanding in the variation in rock strengths across the mining units.
- Determination of Hoek-Brown parameters for numerical modelling (for later stages in the project).

Rock mass classification was conducted on all available geotechnical data that was of a satisfactory quality. Geotechnical data were utilised from 67 geotechnically logged boreholes, of which 61 holes were logged using Bieniawski's (1989) rock mass rating (RMR) system and 6 holes were logged based on Laubscher's (1990) RMR system (Figure 16.7). Based on the results of the rock mass classification, ground control districts were determined.

An Initial geotechnical assessment of the proposed mining method was also carried out.

Figure 16.7 Location of Drillholes

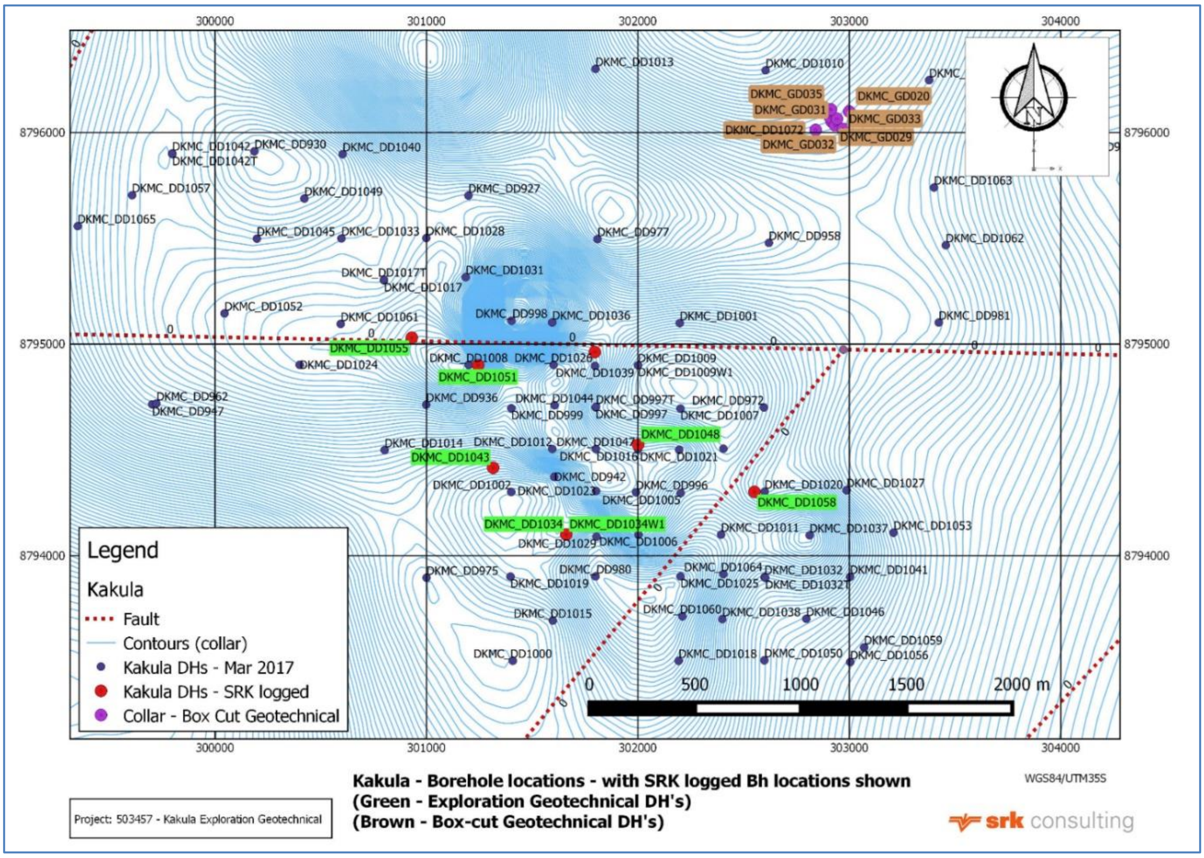


Figure by SRK, 2017

Rock Properties

Rock strength properties have been determined for each mining unit (hangingwall, orebody, and footwall) within the project area based on laboratory testing to provide an understanding of the variation in rock strength across the mining units, to undertake rock mass characterisation and for the determination of Hoek-Brown parameters for non-linear modelling (to be conducted in later stages of the project).

A summary of the laboratory tests results are presented in Table 16.5 and Table 16.6. Note that the unit weight of the various rock types and the strength characteristics of the intact rock material were derived from the UCS and TCS tests. The elastic properties of the intact rock represented by the Young's Modulus and the Poisson's Ratio were derived from the UCM tests where the measurement of deformation during the tests were recorded utilising strain gauges. The mi values for the different mining positions were assessed by fitting the Hoek-brown envelopes to the UCM and TCS laboratory results, as shown in Figure 16.7 to Figure 16.9. It is noted that a limited number of tests were carried out on the orebody and foot wall, which contribute to a lower confidence in the estimation.

Table 16.5 Summary of Laboratory Test Results

Rock Unit		HW	Orebody	FW
Density (kg/m³)	Number of Tests	39	9	15
	Minimum	2.76	2.55	2.54
	Maximum	2.85	2.90	2.80
	Average	2.80	2.81	2.67
	Standard Deviation	0.03	0.11	0.07
UCS (MPa)	Number of Tests	18	6	15
	Minimum	46	7	39.00
	Maximum	142	147	333
	Average	99	69	154
	Standard Deviation	25	52	101
TCS (MPa)	Number of Tests	18	2	1
	Minimum	90	147	245
	Maximum	298	162	245
	Average	187	155	245
	Standard Deviation	59.14	10	0
BFA (°)	Number of Tests	23	5	5
	Minimum	28	33	31
	Maximum	39	35	38
	Average	34	34	35
	Standard Deviation	3	1	3
Brazilian Tensile (MPa)	Number of Tests	30	6	15
	Minimum	3	5	6
	Maximum	18	13	23
	Average	10	8	13
	Standard Deviation	3	3	5

HW = Hangingwall, FW = Footwall.

Table 16.6 Inelastic Material and Hoek-Brown Properties

Rock Unit		HW	Orebody	FW
Young's Modulus (GPa)	Number of Tests	18	6	15
	Minimum	53	32	39
	Maximum	73	68	81
	Average	66	56	68
	Standard Deviation	6	52	101
Poissons Ratio (ν)	Number of Tests	18	6	15
	Minimum	0.21	0.15	0.14
	Maximum	0.34	0.38	0.36
	Average	0.28	0.26	0.22
	Standard Deviation	0.04	0.09	0.06
Hoek-Brown	Sigma ci	86	82	141
	mi	16	15	17

Rock Mass Classification

The rock mass characterisation in this study is based on data that was logged according to two different rock mass rating systems. Data logged on the basis of Bieniawski's (1989) Rock Mass Rating (RMR B89) classification system was used to delineate GCDs because the majority of the drillholes that could be used for data acquisition (61 holes) across the project area were logged based on this system.

Drillholes that were not logged based on Bieniawski's system were logged geotechnically by SRK according to Laubscher's (1990) RMR system (RMR L90). In comparison to RMR B89, RMR L90 is further developed and specially designed for the underground mining environment.

RMR L90 evaluates discrete geotechnical domains based on strength (Intact Rock Strength, IRS), fracture frequency, joint condition and weathering characteristics. Each of the resultant domains is evaluated separately, through the allocation of rating values, within a specific range, for each parameter.

Preliminary rock mass parameters for the design were derived from data that was logged according RMR L90 from 6 drillholes. Based on this data RMR L90, design rock mass strength (DRMS), Q' and geological strength index (GSI) were determined for each mining unit (Table 16.7 and Table 16.8).

Table 16.7 Rock Mass Design Parameters (HW) From SRK Logged Data

	RMR L90	DRMS	Q'	GSI
20^{th} percentile	39	18	12.7	65
50^{th} percentile	44	21	18.8	76
80^{th} percentile	50	24	64.4	81
Average	45	28	-	75

Table 16.8 Rock Mass Design Parameters (Orebody) From SRK Logged Data

	RMR L90	DRMS	Q'	GSI
20^{th} percentile	30	9	6.6	62
50^{th} percentile	38	12	12.8	68
80^{th} percentile	48	16	24.4	75
Average	41	30	-	68

Geotechnical Domains

The delineation of geotechnical domains was conducted by investigating the spatial distribution of the RMR B89 results in and around the orebody. The data available incorporated many more holes than the Laubscher (1990) database and therefore the data covered a much larger area and is more suitable for defining geotechnical domains.

The contour maps in Figure 16.8 and Figure 16.9 show RMR B89 data that is based on either intact rock strength (IRS) ratings from drill core field estimates or uniaxial compressive strengths from laboratory tests. The contour maps are based on the 3% grade shell cut-off which is indicated by black and grey lines. The numbers on the contour lines indicate the thickness of the grade shell.

The maps indicate that the rock mass ratings are higher in the hangingwall than in the orebody. There is also an indication that the rock mass ratings are more desirable in the southern part of the area of interest.

Figure 16.8 RMR B89 Contour Map for the Hangingwall (based on IRS from field estimates)

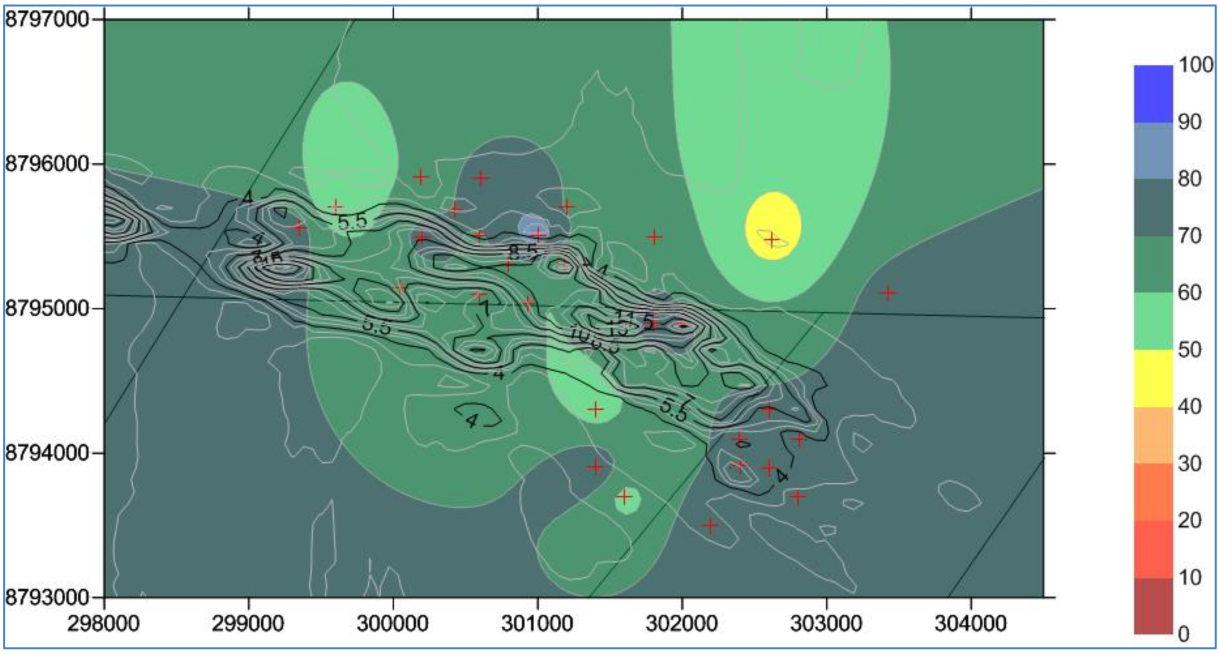


Figure by SRK, 2017

Figure 16.9 RMR B89 Contour Map for the Hangingwall (based on IRS from field estimates)

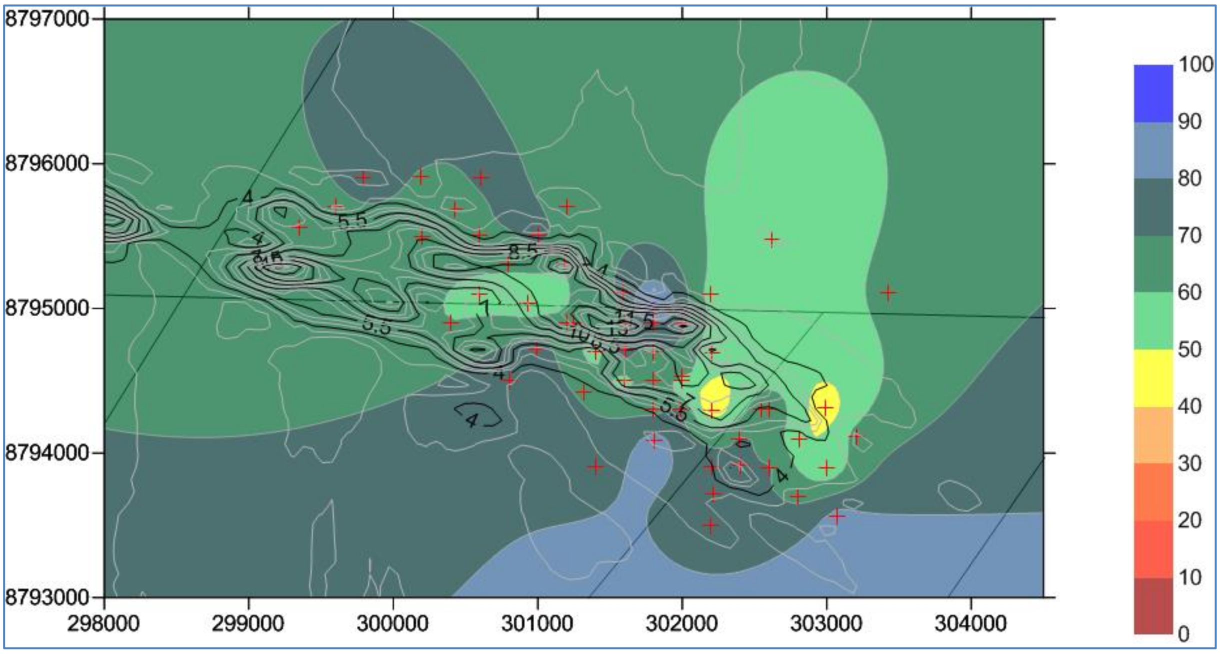


Figure by SRK, 2017

16.1.2.2 Findings and Recommendations

The delineation of geotechnical domains and the derivation of design parameters for design were determined for Kakula based on laboratory testing and rock mass characterisation. Based on the study the following can be concluded:

- The confidence in the geotechnical domains is adequate for the current level of study - but will benefit from further refinement for implementation in subsequent stages of the project.
- Based on the current data, the footwall is the strongest mining unit (σ_{ci} of 141 MPa) compared with the hangingwall (σ_{ci} of 86 MPa) and orebody (σ_{ci} of 82 MPa). The orebody and hangingwall have similar strengths.
- The ground conditions at Kakula are different to Kamoa in that:
  - The UCS values of the diamictite at Kamoa are significantly higher than those at Kakula;
  - The RQD values at Kamoa are lower than at Kakula, which may be in association with the pronounced weathering at Kamoa; and
  - The interbedded pyritic sandy siltstone (KPS) present at Kamoa is absent at Kakula.

The recommendations for the next phase of study are summarised as follows:

- Additional rock mass characterisation of boreholes is required to obtain better resolution across the orebody.
- Structural data should be collated to determine structural domains and joint set orientations and characteristics.
- Quality control and quality assurance of data collection must be strictly carried out.
- Additional footwall and triaxial tests are recommended for the orebody as well as for the footwall to reduce uncertainty in terms of rock mass strength and help to accurately define Hoek-Brown rock mass strength parameters. The selection of specimens for these tests should be carried out against well-defined boundaries of the orebody.
- All additional geotechnical data gathered following from the project should be incorporated into the database and evaluated.
- The work on defining geotechnical domains from this current study should be refined in the next phase of the project as additional data is gathered.
- This information will be used for the geotechnical design in the next phase of study.

16.2 Underground Mining

The mining methods for the Kamoa deposit have been modified from previous studies. The current main mining methods include room-and-pillar for the mineralised zones above the 150 m depth and controlled convergence room-and-pillar below the 150 m depth.

The Kansoko Mine will be a mobile, trackless mining operation. Access to the mine is planned to be via a twin decline system from Kansoko Sud portal.

The Kamoa 2017 PFS production schedule has been developed based upon a 6 Mtpa production rate and high-grade scenario.

The overall mineral reserve could have been larger based on the size of the Kamoa deposit; however, only the targeted best 150.5 Mt in the Centrale and Sud regions were evaluated. From the targeted resource, a mine schedule that produced 125.2 M tonnes was produced.

16.2.1 Mining Methods

The Kamoa orebody geometry indicates different orebody thicknesses and slopes. The orebody dips between 0° and 35°, with an average dip of 17°. The thickness varies between 3.0 m and 6.0 m.

Access to the mine is via a twin declines system from the Kansoko Sud Portal, to support mining the Kamoa deposit. This has a total of 11,736 m of development. Besides primary development, the two mining methods for this orebody are room-and-pillar mining and controlled convergence room-and-pillar mining.

16.2.1.1 Room-and-Pillar

Room-and-pillar mining will be used in the mineralised zone between 60–150 m, to minimise the risk of surface subsidence. Continuing room-and-pillar mining below 150 m is required in selected areas for production ramp-up.

The production development of the room-and-pillar method will be in a grid-like fashion. The mining areas were divided to distinguish between the geotechnical needs for the room-and-pillar design above and below 150 m from surface elevation. The room development will run parallel to the strike of the panel for dips less than 20°, with belt drives running at an acute angle to the room drifts, to ensure the grade of the production drifts remains at or below minimum specifications. Where the dip is greater than 20°, the rooms will be developed slightly off the strike, to accommodate the acute angle between the room development and the belt drives. A typical room-and-pillar mining panel is demonstrated in Figure 16.10.

Figure 16.10 Typical Room-and-Pillar Mining Panel

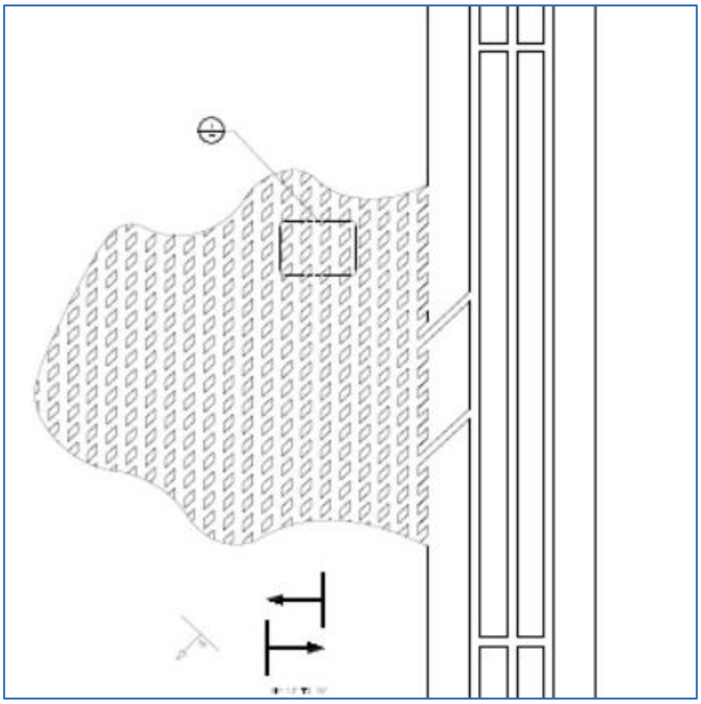


Figure by Stantec, 2017.

Long-term stability is required for room-and-pillar to allow access for the miners while in production as the mining front begins at the access and progresses toward the ends of the panel. These room-and-pillar areas, designed to prevent subsidence, will remain accessible if maintained and ventilated.

Referring to Table 16.3, the extraction ratios for room-and-pillar mining were based on the dip and the height of the panels and Cuprum's resulting pillar design.

Figure 16.11 shows a proposed typical layout at a depth of 100 m and an apparent dip of 8°.

At this depth, pillars are required to be 9 m x 9 m with 10 m-wide rooms. A mining height of 6 m has been assumed. Eight (8) rooms will be required to meet the maximum panel span of 152 m, which will be bounded by 20 m regional pillars. A maximum strike length of 504 m has been allowed. A row of regional pillars 15 m wide will then be required before the next panel is started.

The calculated extraction ratios for areas mined using the stepped room-and-pillar method vary with depth. These are summarised in Table 16.9 for orebody thickness of 4 m and 6 m.

Figure 16.11 Stepped Room-And-Pillar at 100 m Below Surface and 6 m Mining Height

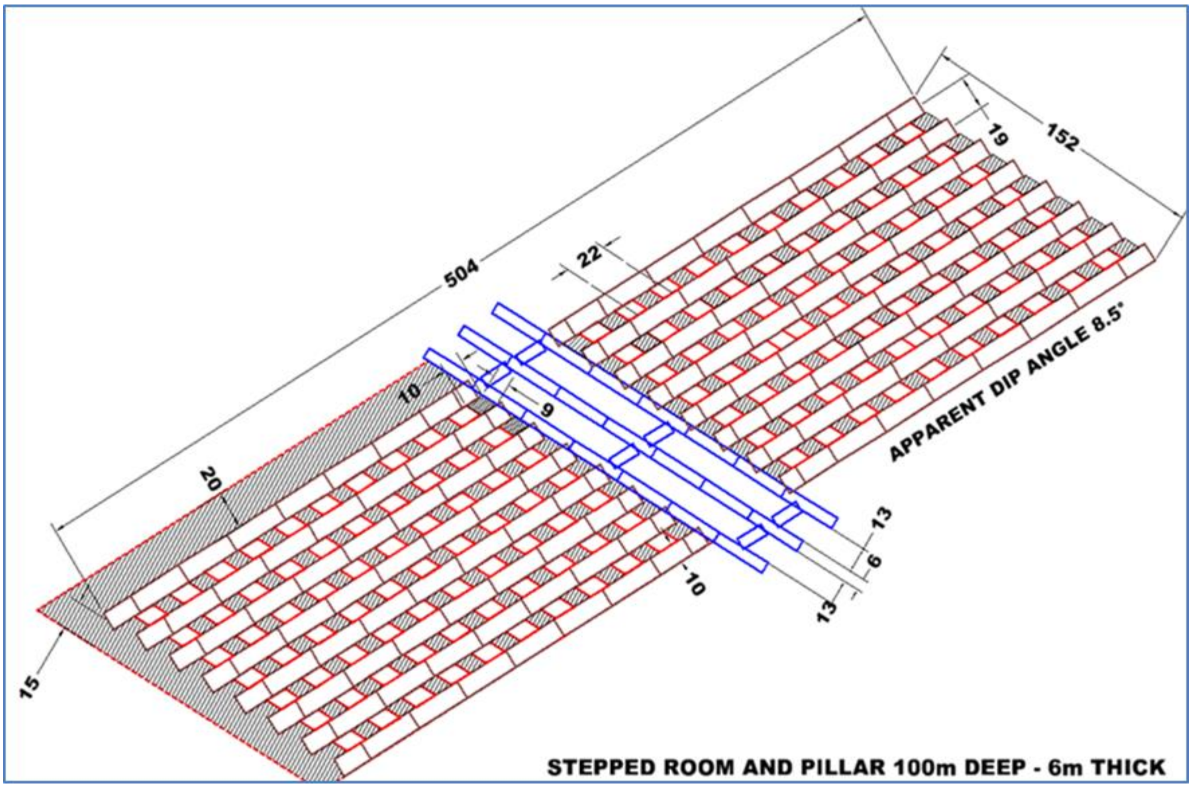


Figure by Kamoa Copper SA, 2015.

Table 16.9 Calculated Extraction Ratios for Stepped Room-And-Pillar Mining

Depth (m)	Ore Thickness (m)	Primary Extraction		Final Extraction	
		In-panel FOS = 1.5		Slashing In-panel to FOS = 1.2	
		In-panel Extraction (%)	Total Primary Extraction (%)	In-panel Extraction (%)	Final Extraction (%)
100	6.0	80	68	82	70
100	4.0	87	76	87	76
200	6.0	77	66	81	69
200	4.0	81	70	84	73
300	6.0	69	56	77	62
300	4.0	75	61	78	64
400	6.0	63	51	71	57
400	4.0	69	56	77	63
500	6.0	58	44	69	53
500	4.0	66	49	71	53

16.2.1.2 Controlled Convergence Room-and-Pillar

Controlled convergence room-and-pillar mining will be used in the mineralised zone below 150 m. An initial panel will be taken as a trial panel to confirm the method viability. Once 70% complete, the additional panels will start production mining.

The development of the panel requires secondary drifts to be excavated on the perimeter, to allow access of equipment to the production headings. This secondary development will consist of two headings connected by cross-cuts. If the panel is being mined from the extents toward the access, then the secondary development will be driven completely around the perimeter of the panels. The panel dimensions are generally 300 m wide and a minimum of 500 m long, where possible. In the case where the mining front is progressing away from the access, the perimeter development will only be designed along the sides of a panel. If truck haulage is required for the panel, the secondary development will be large enough to allow trucks to be driven into the panels. Figure 16.12 is a typical controlled convergence room-and-pillar panel, with a mining direction advancing toward the access.

Upon completion of the required secondary development, production development will begin by establishing room drifts and their associated belt drifts. Similar to room-and-pillar mining, the angle between these drifts are determined by the dip and thickness of the orebody. The angles will accommodate the maximum gradient permissible in this design, which is 12° or less.

During the retreat of a final panel where a panel adjacent has been mined out, the belt drives will carry into the secondary development nearest to that panel as part of the mining front.

The room and belt drives will form technological pillars. These pillars are designed to compress the load of the working back. As the mining working area increases, the pillars take more stress and cause the convergence. It has been determined that the maximum mining area to activate is no more than three-belts distance from the working face to the pillar scraping.

These technological pillars will be reduced through scraping or drill and blast to a remnant pillar size as defined in Figure 16.12. The remnant size is the minimum size pillar that is allowed to be scraped while still ensuring the overall front is supported and continues to converge. Once remnant size pillars are established, personnel and equipment will be prevented from working in that area.

Figure 16.12 Typical Controlled Convergence Room-and-Pillar Mining Panel

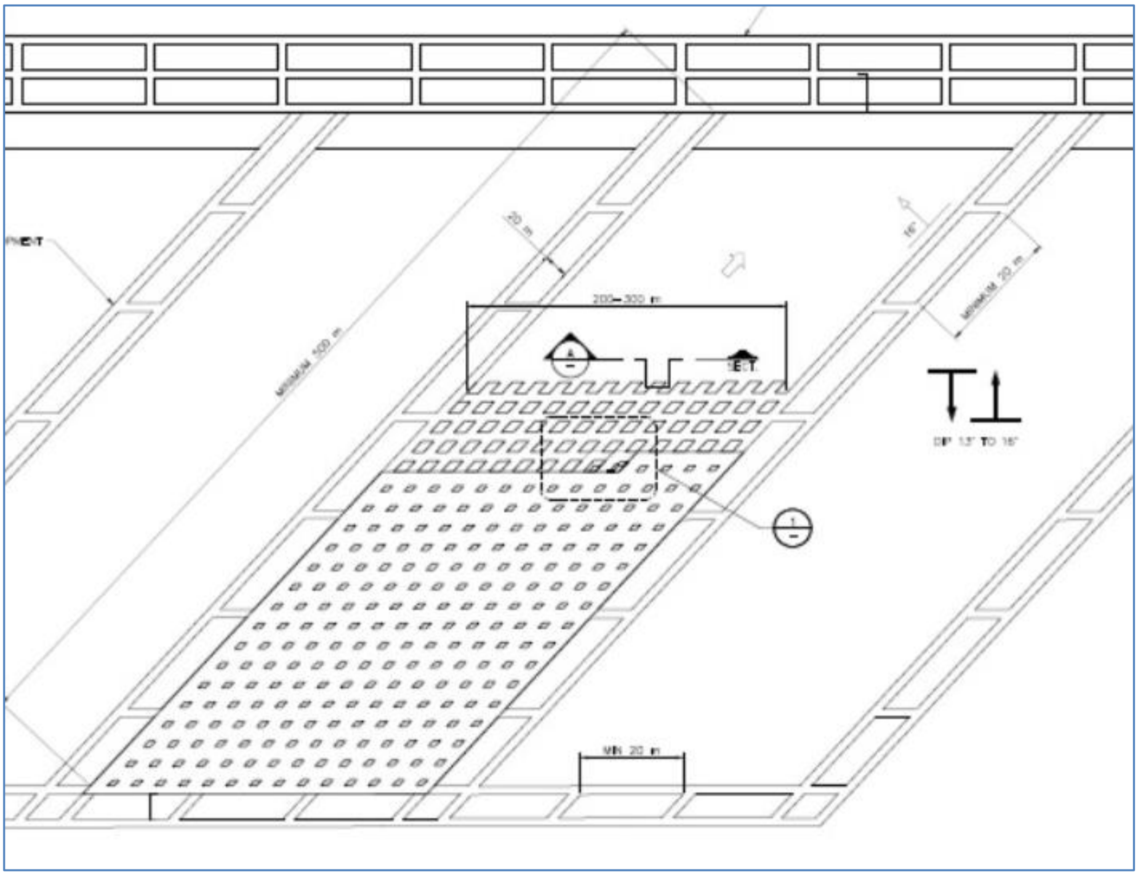


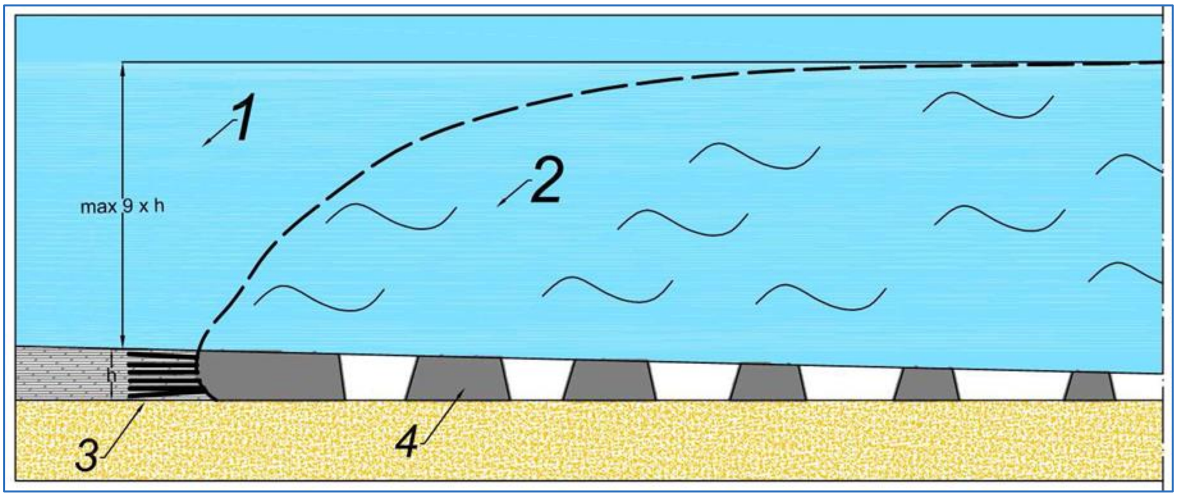
Figure by Stantec, 2017.

Controlled convergence room-and-pillar mining is currently utilised by KGHM at their mines in Poland. It is based on the strength and strain parameters of the rock that make up the mining panel supporting pillar or technological pillars and includes the following parameters:

- Ore zone depths below 150 m.
- Strength of the immediate roof (i.e. roof bolting and handling of the rock burst threat).
- Strength and strain parameters of the rocks within the roof of the extraction panel (i.e. the slow bending above the extraction space and in the workings).
- Technological pillars (pillars between rooms) designed to work in the post-destruction strength state to maximise ore extraction.
- Cuprum has developed the controlled convergence room-and-pillar methodology at its mines in Poland and are the technical contributors to its adaptation for the Project.

Extraction mining with roof deflection and pillar strength in the post-destructive state is based on a modified Labasse hypothesis (1949) (see Figure 16.13). The relationship between the pillar height-to-width ratio should be within the range of 0.5–0.8. This ensures the progressive transition of the technological pillars into the post-destructive strength state, enabling a smooth roof-bending strata (destressed and delaminated rock mass) above the workings.

Figure 16.13 Controlled Convergence Room-and-Pillar Rock Mass Impact



1 – rock mass prior to extraction; 2 – destressed and delaminated rock mass; 3 - blasting holes; 4 - primary pillars.
Figure by KGHM Cuprum, 2014.

Figure 16.14 and Figure 16.15 are the mining panels in the Centrale and Sud zones.

Figure 16.14 Centrale Mine Design – Plan View

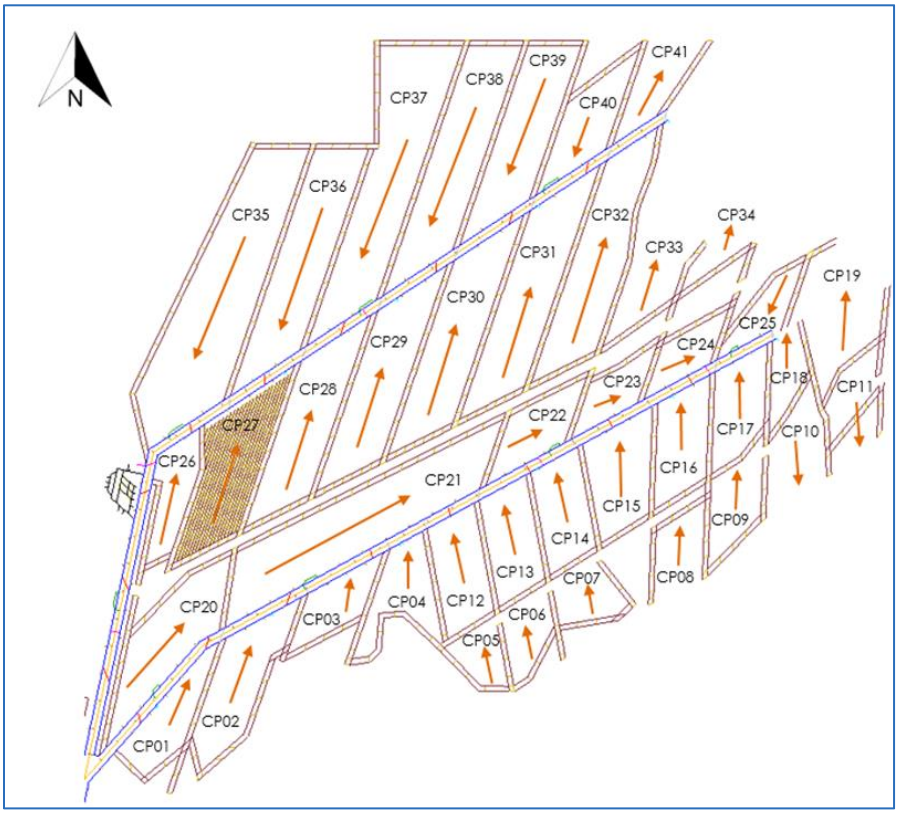


Figure by Stantec, 2017.

Figure 16.15 Sud Panel Number and Mining Direction

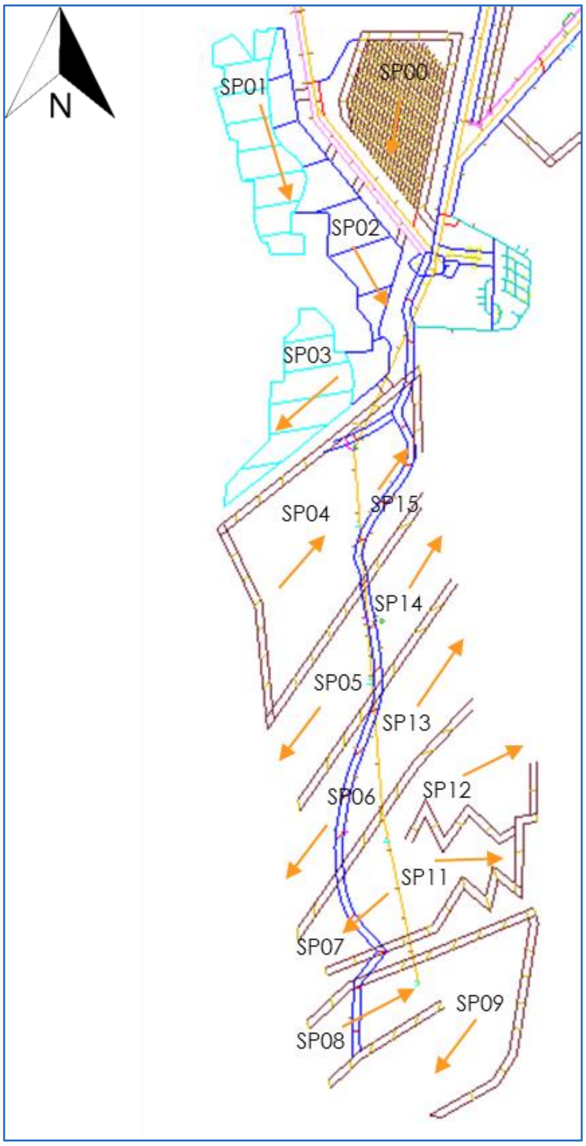


Figure by Stantec, 2017.

Referring to Table 16.4, the extraction ratios for controlled convergence room-and-pillar mining were based on the dip and the height of the panels and Cuprum's resulting pillar design.

16.2.2 Mining Dilution and Recovery Factors

16.2.2.1 Dilution Grade

To obtain dilution grades, three dilution shells were constructed around each production panel shape to report the grade and density outside of the targeted resource. The three shells comprise a 1.0 m hangingwall (HW1) dilution shell on top of the production panel shape and 2 x 1.0 m footwall (FW1, FW2) dilution shells on the bottom. The block model interrogated the dilution shells by block centre to provide the dilution grades and densities for each shell.

The grades were then applied to the calculated tonnage of dilution for each production shape. Figure 16.16 illustrates a section view of the production panel shape and the corresponding dilution shells.

Figure 16.16 Production Panel with Dilution Shells

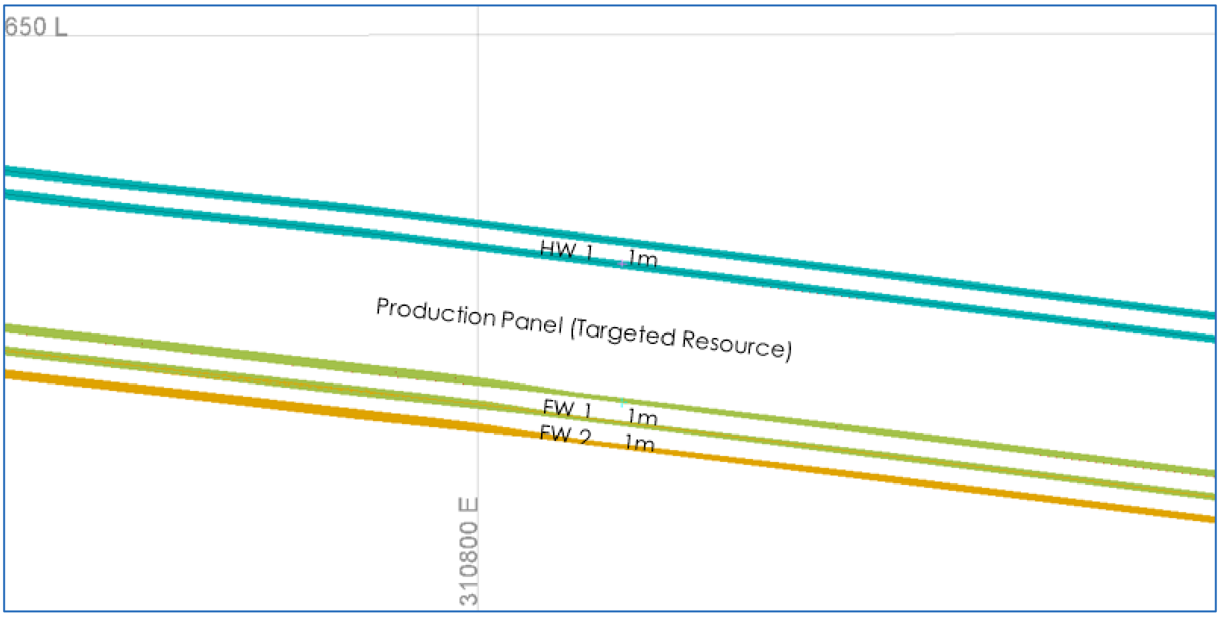


Figure by Stantec, 2017.

16.2.2.2 Dilution Tonnes

Primary Development

Primary development includes the two service drifts with a cross-sectional dimension of 5.5 m wide x 6.0 m high and a conveyor drift with a cross-sectional dimensions of 7.0 m wide x 6.0 m high.

As shown in Figure 16.17, these development headings were assumed to have a flat back with arched corners and will not be affected by the height or spatial location of the grade shell, which would result in a fluctuating grade and planned dilution percentage in each segment for the length of the drift. Back and wall dilution is assumed to have an average overbreak of 0.1 m. Table 16.10 summarises the calculated overbreak dilution for the development headings.

Figure 16.17 Typical Primary Development Drift

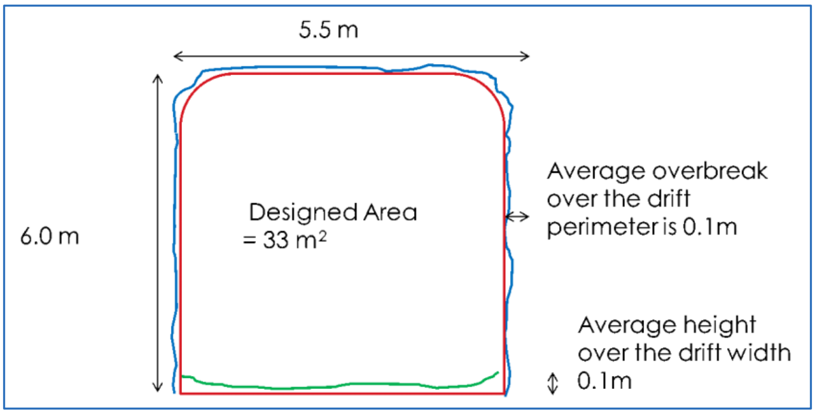


Figure by Stantec, 2017.

Table 16.10 Primary Dilution

Heading Size	Dilution (%)	HW Dilution (m)	Wall Dilution per Side (m)
5.5 m wide x 6.0 m high	5.4	0.1	0.1
7.0 m wide x 6.0 m high	4.6	0.1	0.1

Room-and-Pillar

Room-and-pillar production only includes production from the room-and-belt drifts. There is no pillar extraction for this method.

For hangingwall dilution, overbreak is assumed to average 0.15 m. No dilution from the walls was considered, since the pillar width must be maintained. Controlled blasting practices will be required to ensure that the walls are broken to design width.

The footwall dilution is a planned dilution and is based on the dip and thickness of the production panel shape. Figure 16.18 illustrates a typical drift shape for 13°–16° dip and a 4.5 m thickness.

Figure 16.18 Typical Room-and-Pillar Room Drift Shape (13°–16° Dip, 4.0–5.0 m Height)

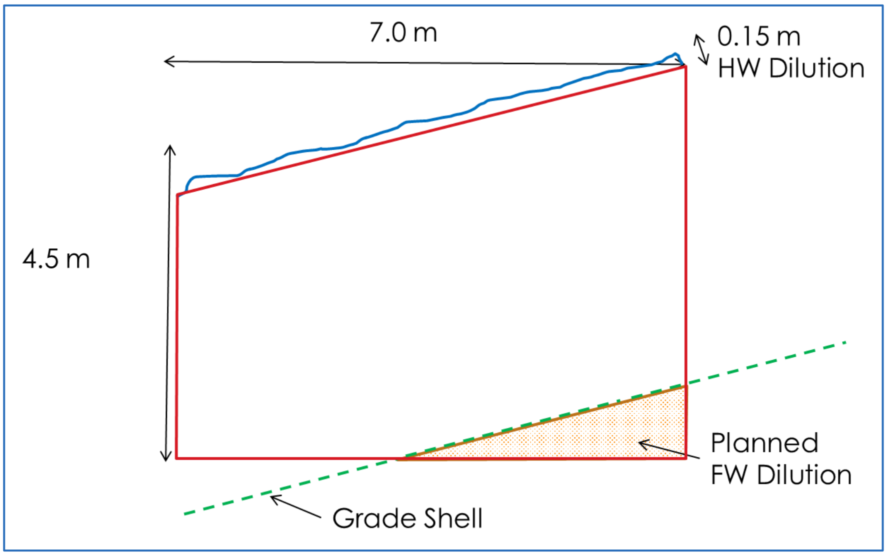


Figure by Stantec, 2017.

Table 16.11 lists each of the in-panel ore development shape types used for the Kamoa deposit. For thicknesses where the short side wall is less than 2.5 m high, the angle of the back and the floor are adjusted. The result is a slightly increased footwall dilution and slightly reduced hangingwall dilution. This occurs for all 3.0 m high thicknesses with a dip greater than 12° and all 3.5 m high thicknesses with a dip greater than 16°.

Table 16.11 Room-and-Pillar Ore Development Dilution Percentages

Dip (degrees)	Ore Thickness (m)	Footwall Dilution (%)	Hangingwall Dilution (%)
0–12	3.0	2.9	4.8
13–16	3.0	7.0	4.6
17–20	3.0	9.1	4.5
21–25	3.0	10.0	4.4
26–30	3.0	10.4	4.4
31–35	3.0	11.2	4.3
0–12	3.5	2.4	4.1
13–16	3.5	5.2	4.2
17–20	3.5	6.5	4.2
21–25	3.5	8.0	4.1
26–30	3.5	8.8	4.1
31–35	3.5	9.5	4.1
0–12	4.5	1.8	3.2
13–16	4.5	3.8	3.3
17–20	4.5	4.5	3.3
21–25	4.5	5.3	3.4
26–30	4.5	5.7	3.5
31–35	4.5	6.3	3.6
0–12	5.5	1.5	2.7
13–16	5.5	2.9	2.7
17–20	5.5	3.3	2.8
21–25	5.5	3.8	2.8
26–30	5.5	4.0	2.9
31–35	5.5	4.1	3.0
0–12	6.0	1.3	2.4
13–16	6.0	2.5	2.5
17–20	6.0	2.9	2.5
21–25	6.0	3.2	2.6
26–30	6.0	3.3	2.6
31–35	6.0	3.4	2.8

Controlled Convergence Room-and-Pillar

Controlled convergence room-and-pillar production includes the development associated with panel perimeter drifts (secondary development) and production from the room-and-belt drifts plus pillar extraction.

For hangingwall dilution, overbreak is assumed to average 0.15 m. No dilution from the walls was considered since the overbreak is mainly from pillars, which will be extracted later in the production cycle. The hangingwall overbreak dilution is expected to project across the back of the pillar as it is extracted.

The footwall dilution is a planned dilution and is based on the dip and thickness of the production panel shape. Figure 16.19 illustrates a typical drift shape for a 13°–16° dip and a 4.5 m thickness.

Figure 16.19 Typical Controlled Convergence Room-and-Pillar Room Drift Shape (13°–16° Dip, 4.0–5.0 m Height)

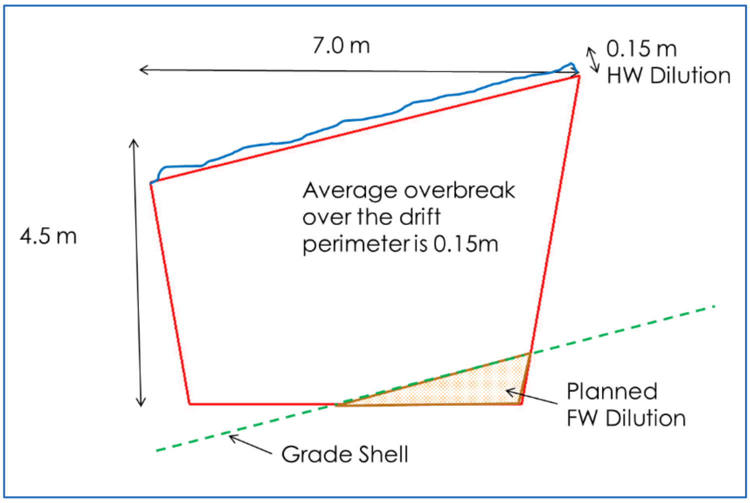


Figure by Stantec, 2017.

Table 16.12 lists each of the in-panel ore development shape types used for the Kamoa deposit. Ore from the secondary panel drifts was captured in the extraction ratio tonnage calculations and therefore was not counted in the development tonnage.

Table 16.12 In-Panel Ore Development Dilution Percentages

Dip (degrees)	Ore Thickness (m)	Footwall Dilution (%)	Hangingwall Dilution (%)
0–12	3.0	2.2	5.15
13–16	3.0	5.4	5.03
17–20	3.0	7.2	4.91
21–25	3.0	7.3	4.79
26–30	3.0	7.4	4.74
31–35	3.0	7.3	4.61
0–12	3.5	1.7	4.51
13–16	3.5	3.5	4.60
17–20	3.5	4.2	4.64
21–25	3.5	5.2	4.61
26–30	3.5	5.7	4.58
31–35	3.5	5.5	4.52
0–12	4.5	1.2	3.64
13–16	4.5	2.1	3.72
17–20	4.5	2.4	3.79
21–25	4.5	2.6	3.90
26–30	4.5	2.5	3.98
31–35	4.5	2.3	4.17
0–12	5.5	0.8	3.08
13–16	5.5	1.3	3.15
17–20	5.5	1.4	3.21
21–25	5.5	1.3	3.31
26–30	5.5	1.2	3.38
31–35	5.5	0.9	3.55
0–12	6.0	0.7	2.88
13–16	6.0	1.1	2.94
17–20	6.0	1.1	3.00
21–25	6.0	0.9	3.09
26–30	6.0	0.8	3.16
31–35	6.0	0.5	3.32

In secondary development where the back height is restricting truck access, a decision was made to increase the back height, which increases the amount of dilution associated with the hangingwall. The truck height for this evaluation was 3.86 m from the sill to the top of the fully loaded truck. The restriction occurs for drift heights of 3.5 m and lower. The resulting hangingwall and footwall percentages are summarised in Table 16.13.

Table 16.13 Increased Panel Height Dilution Percentages

Dip (degrees)	Ore Thickness (m)	Footwall Dilution (%)	Planned Hangingwall Dilution (%)	Unplanned Hangingwall Dilution (%)
0–12	3.0	2.2	23.5	4.0
13–16	3.0	5.4	22.8	3.9
17–20	3.0	7.2	22.1	3.9
21–25	3.0	7.3	21.0	3.8
26–30	3.0	7.4	20.4	3.8
31–35	3.0	7.3	19.0	3.8
0–12	3.5	1.7	10.0	4.1
13–16	3.5	3.5	10.2	4.2
17–20	3.5	4.2	10.2	4.3
21–25	3.5	5.2	9.2	4.2
26–30	3.5	5.8	9.5	4.2
31–35	3.5	5.5	9.8	4.1

16.2.2.3 Mining Recovery

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

Primary Development

For primary development, the recovery is 98%. Lost tonnage is a result of losses due to the corners of the drift and muck that settle into the irregularities in the floor. Stantec estimated 0.1 m of rock material will be lost on the floor. See Figure 16.19.

Figure 16.20 Recovery from a Primary 5.5 m W x 6.0 m H Heading

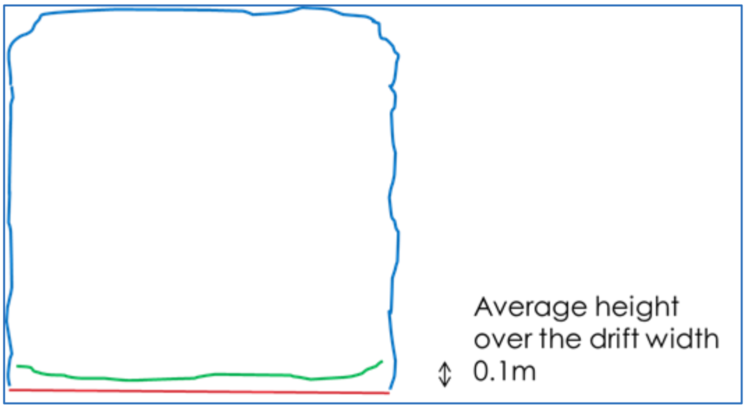


Figure by Stantec, 2017.

Room-and-Pillar

Room-and-pillar mining is development intensive and will have recoveries similar to primary development. Some material will be left along the corners of the walls but will be recovered during the pillar extraction phase, so a recovery of 98% is expected. The recovery due to the losses in the production drifts are illustrated in Figure 16.21.

Figure 16.21 Recovery Losses in a Typical Room-and-Pillar Production Drift

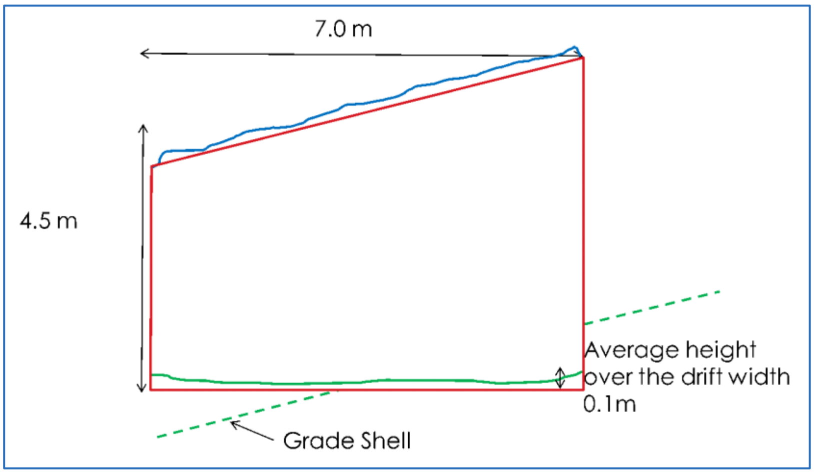


Figure by Stantec, 2017.

Controlled Convergence Room-and-Pillar

Controlled convergence room-and-pillar mining is similar to room-and-pillar mining as it is development intensive and will also have recoveries similar to primary development. Some material will be left along the corners of the walls but will be recovered during the pillar extraction phase, so a recovery of 98% is expected. The recovery due to the losses in the production drifts are illustrated in Figure 16.22.

Figure 16.22 Recovery Losses in a Typical Controlled Convergence Room-and-Pillar Production Drift

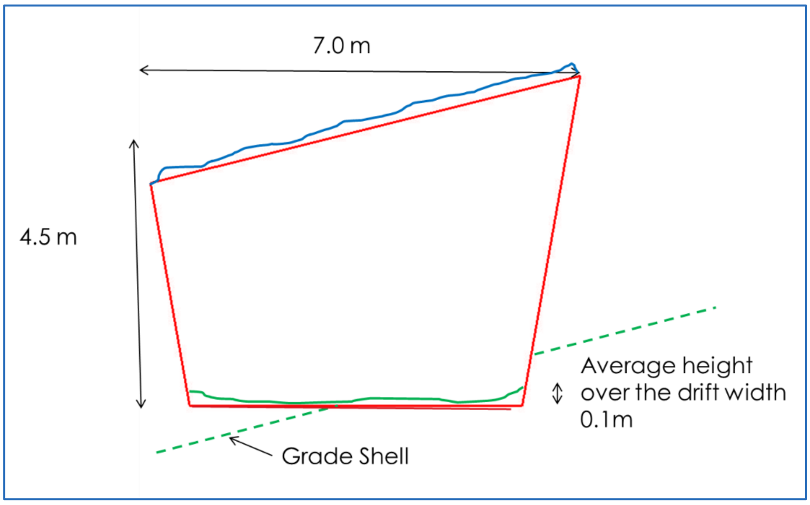


Figure by Stantec, 2017.

Due to the space of the working area around the pillar and the larger muck size created from the scraping process of extraction, the initial recoveries will remain high. Ore extraction losses will occur when pillars cannot be completely recovered due to deterioration of ground conditions and steep dips. Based on these factors and experience, a 95% mining recovery was applied to pillar extraction tonnages.

16.2.3 Mining Access Design

Mine access is required to ensure safe and reliable transport of mining personnel and equipment, for production, for intake and exhaust ventilation-ways, and to facilitate the reticulation of all services to and from the mine workings.

Key access design objectives were to:

- Access the workings in a way that minimises capital development.
- Facilitate an aggressive production build-up, targeting the high-grade areas as quickly as possible.

Access into the mine will be via a set of twin declines from the portal down to the Kansoko Sud/Centrale breakaway. One decline will house the main conveyor and the other will be used as the service decline. The declines from the surface will be inclined at -8.5°, considered the optimal inclination for mechanised equipment.

The conveyor decline will extend beyond the Kansoko Sud/Centrale breakaway to the storage silo system. The conveyor decline inclination will increase to -12° to allow construction of the storage silos below the ore horizon.

From the top of the storage silo system, the Kansoko Sud conveyor decline will be developed to the south to the most southerly Kansoko Sud mining block.

The service decline will terminate at the Kansoko Sud/Centrale breakaway, and a set of triple declines will be developed down the Kansoko Sud/Centrale access to the breakaway of the Kansoko Sud roadway.

Triple declines will then be developed into the Centrale North and South mining areas, and a twin roadway system will develop into the Kansoko Sud mining area.

Development dimensions will be 5.5 m (W) x 6.0 m (H) for the service drift and 7.0 m (W) x 6.0 m (H) for the conveyor drift, based on the conveyor design, ventilation intake requirements, and sizes of equipment.

The portal is positioned to facilitate quick access to the shallower parts of the orebody and to the higher-grade areas of the Kansoko Sud mining area. It also allows early development towards the high-grade areas of the Centrale mining area. Figure 16.23 shows the portal, declines, and underground infrastructure.

Figure 16.23 Underground Access Infrastructure

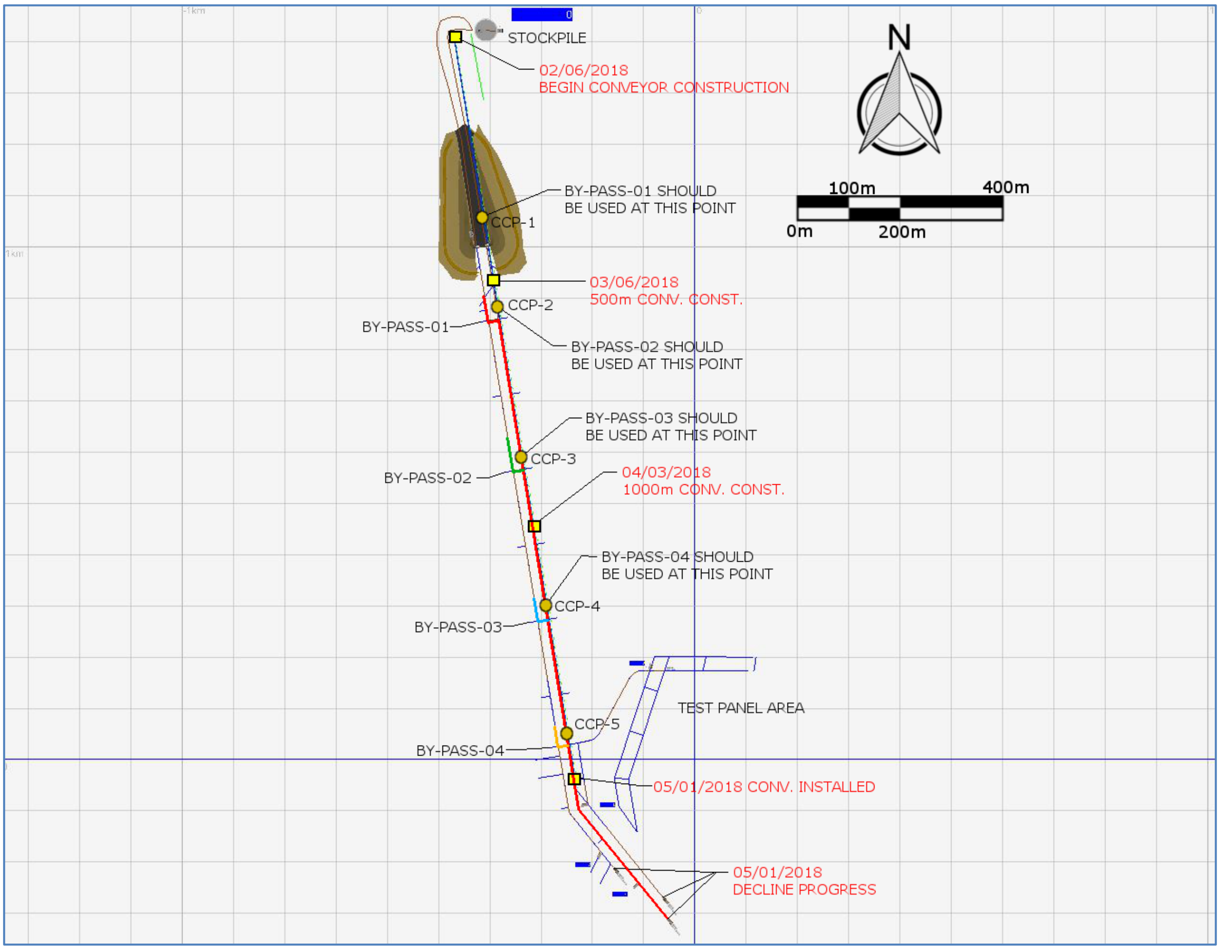


Figure by Kamoa Copper SA, 2015.

Figure 16.24 shows the position of the portal in relation to the surface infrastructure.

Figure 16.24 Portal Position in Relation to Surface Infrastructure

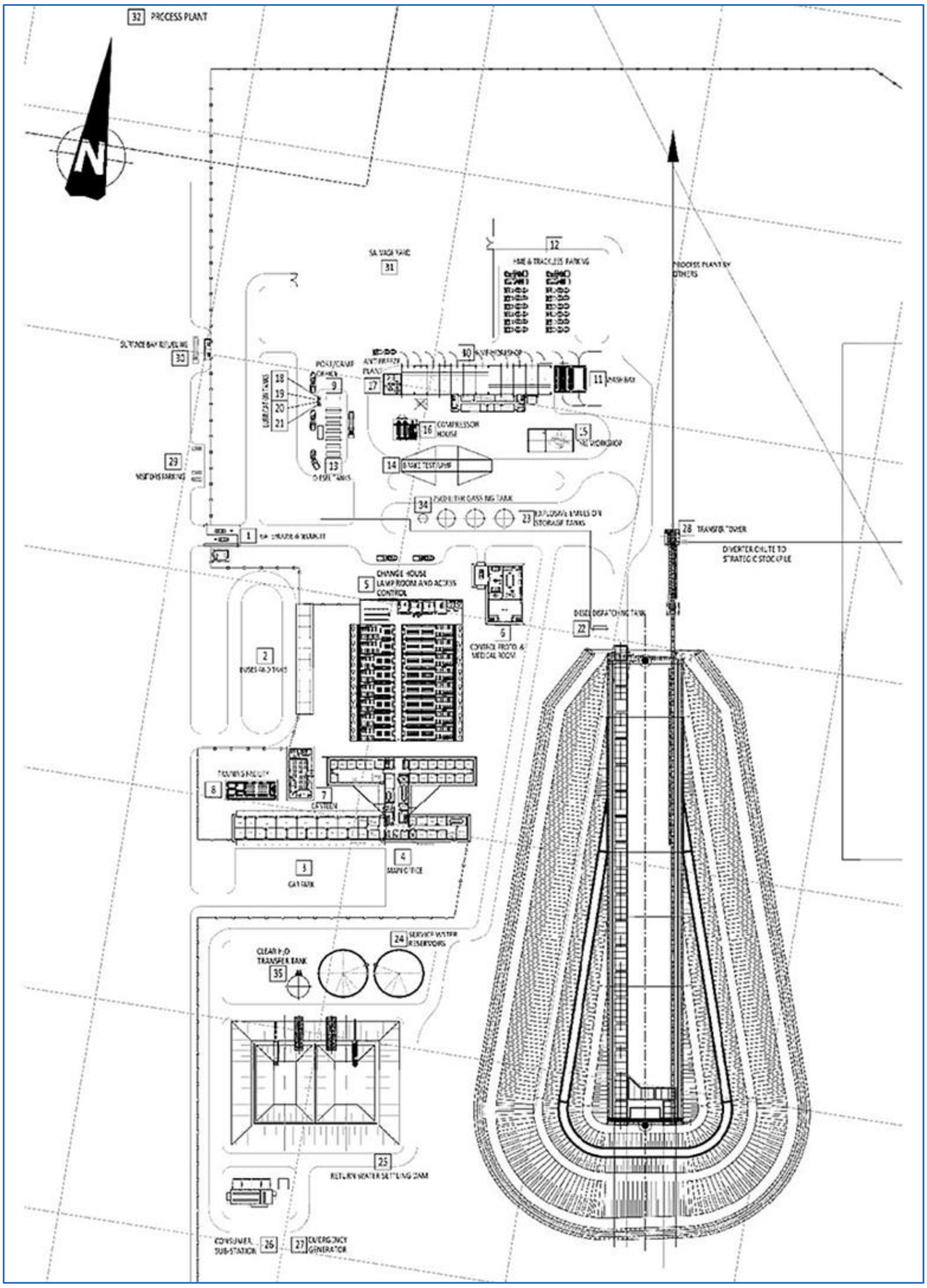


Figure by Kamoa Copper SA, 2015.

16.2.4 Mining Schedule

16.2.4.1 Development and Construction Schedule

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

Mine development will occur in the following three main phases:

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

Table 16.14 summarises the LOM development and production.

Table 16.14 LOM Development and Production Summary

Waste Development	
Lateral Development (m)	31,390
Lateral Development Tonnes (t)	3,297,172
Mass Excavation Lateral Equivalent (m)	2,517
Mass Excavation Tonnes (t)	224,276
Vertical Development(m)	5,182
Vertical Development Tonnes (t)	296,952
Total Waste Development	
Meters (m)	39,089
Tonnes (t)	3,818,399
Production by Mining Method	
Ore Development Meters (m)	114,205
Ore Development Tonnes (t)	10,665,176
Room-and-Pillar Production (m)	36,743
Room-and-Pillar Production (t)	3,396,830
Controlled Convergence Room-and-Pillar Production (m)	813,559
Controlled Convergence Room-and-Pillar Production (t)	111,120,408
Total Ore Production	
Total Ore Development (m)	114,205
Total Production (m)	850,302
Total Tonnes (t)	125,182,414
Diluted Grade	
NSR ($/t)	$168.06
TCu (%)	3.81
AsCu (%)	0.32
S (%)	2.49
As (%)	0.00
Fe (%)	6.14
Density (t/m^3)	2.93

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

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

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

16.2.4.2 Mine Development Plan and Scheduling

Productivity Rates

For primary development, the rates in Table 16.15 were calculated using first principles. Cycle inputs were obtained from various sources (e.g. original equipment manufacturer [OEM], external consultants, specialists) and compared with Stantec inputs. The cycles were updated accordingly following team discussions. The shift rotation forms the basis of the zero-based rate calculations, which is summarised in Table 16.16.

Table 16.15 Primary Development Rates

Description	Single-Heading Performance (m/d)	Double-Heading Performance (m/d)	Multi-Heading Performance (m/d)	Single-Heading Performance (t/d)	Double-Heading Performance (t/d)	Multi-Heading Performance (t/d)
5.5 W x 6.0 H – Semi-Arch (Service Drifts)	3.96	5.35	5.94	395	533	592
5.5 W x 6.0 H – Flat (Cross-cut Drift)	3.93	5.31	5.89	389	525	584
7.0 W x 6.0 H – Semi-Arch (Conveyor Drifts)	3.49	N/A	N/A	427	N/A	N/A

Table 16.16 Shift Rotations and Calculations

Shift Cycle	Calculations
Days per Year	360 d
Number of Crews	4
Shifts per Day	2
Shift Duration	12 h
Traveling Time – In	31.5 min
Traveling Time – Out	31.5 min
Lunch	60 min
Pre-Shift Safety Meeting and Pre-Shift Inspections	15 min
Actual Face Time per Shift	582 min
Actual Face Time per Day	1,164 min
Effective Working Time per Hour	83.3%
Effective Face Time per Shift	8.08 h
Effective Face Time per Day	16.17 h

Excludes surface raiseboring and current decline contractor effective working hours.

The zero-based rate calculations for secondary and production drift development in controlled convergence room-and-pillar mining were developed based on the drift cross-sections that have a plan view width of 7.0 m, with wall slopes 10° from vertical. Room-and-pillar mining also has a width of 7.0 m but has vertical walls. Secondary drifting is not required for room-and-pillar mining. Secondary and production drifts generally follow the ore, with minor additional cross-section enlargement into the waste of the back and floor for thin, steeply dipping areas.

For drift heights less than 4.0 m at the centre of the drift, the back height will be increased to accommodate a haul truck where required.

Table 16.17 details secondary drift cycle times. The crew production drift cycle times are detailed in Table 16.18.

Table 16.17 Secondary Drift Cycle Times

Drift Type	Dimension (Centre height)	Description (Ore dip)	Single-Heading Performance (m/d)	Double-Heading Performance (m/d)	Multi-Heading Performance (m/d)	Single-Heading Performance (t/d)	Double-Heading Performance (t/d)	Multi-Heading Performance (t/d)
5	7.0 W x 3.0 H	≤21°	5.80	7.84	8.71	366	494	549
6	7.0 W x 3.5 H	≤21°	5.47	7.38	8.20	397	536	596
7	7.0 W x 4.0 H	≤21°	5.18	6.99	7.77	421	568	631
8	7.0 W x 4.5 H	≤21°	5.38	7.27	8.07	504	680	755
9	7.0 W x 5.0 H	≤21°	4.97	6.71	7.46	508	686	762
10	7.0 W x 5.5 H	≤21°	4.76	6.43	7.14	542	732	813
11	7.0 W x 6.0 H	≤21°	4.61	6.23	6.92	565	763	848
12	7.0 W x 3.0 H	>21°–25°	4.46	6.03	6.70	285	385	427
13	7.0 W x 3.5 H	>21°–25°	4.45	6.01	6.68	300	405	450
14	7.0 W x 4.0 H	>21°–25°	4.28	5.78	6.43	303	410	455
15	7.0 W x 4.5 H	>21°–25°	4.41	5.95	6.61	354	478	531
16	7.0 W x 5.0 H	>21°–25°	4.28	5.78	6.42	369	498	554
17	7.0 W x 5.5 H	>21°–25°	4.04	5.46	6.07	385	520	577
18	7.0 W x 6.0 H	>21°–25°	3.84	5.18	5.76	393	530	589
34	7.0 W x 3.0 H	≥25°–30°	4.35	5.87	6.52	284	383	426
35	7.0 W x 3.5 H	≥25°–30°	4.25	5.74	6.38	296	399	443
36	7.0 W x 4.0 H	≥25°–30°	4.05	5.47	6.08	296	400	444
37	7.0 W x 4.5 H	≥25°–30°	4.19	5.66	6.29	338	456	506
38	7.0 W x 5.0 H	≥25°–30°	4.10	5.54	6.16	351	474	527
39	7.0 W x 5.5 H	≥25°–30°	3.87	5.23	5.81	369	499	554
40	7.0 W x 6.0 H	≥25°–30°	3.72	5.02	5.58	381	514	571
48	7.0 W x 4.0 H	≥30°–35°	3.94	5.32	5.92	300	405	450
49	7.0 W x 6.0 H	≥30°–35°	3.67	4.95	5.50	377	508	565

Table 16.18 Production Drift Cycle times (Crews for Controlled Convergence Room-and-Pillar)

Drift Type	Dimension (Centre height)	Description (Ore dip)	Face Drilling Performance (h/t)	LHD Mucking Performance (h/t)	Bolting Performance (h/t)	Critical Activity	Critical Activity Performance (t/d)	Critical Activity Performance (m/d)
19	7.0 W x 3.0 H	≤20°	0.011	0.011	0.009	Face Drilling	1447	23
20	7.0 W x 3.5 H	≤20°	0.010	0.011	0.008	LHD Mucking	1503	21
21	7.0 W x 4.0 H	≤20°	0.010	0.011	0.007	LHD Mucking	1489	19
22	7.0 W x 4.5 H	≤20°	0.009	0.006	0.007	Face Drilling	1836	20
23	7.0 W x 5.0 H	≤20°	0.008	0.006	0.007	Face Drilling	1925	19
24	7.0 W x 5.5 H	≤20°	0008	0.006	0.006	Face Drilling	2076	18
25	7.0 W x 6.0 H	≤20°	0.008	0.006	0.006	Face Drilling	2146	18
26	7.0 W x 3.0 H	>20°–25°	0.011	0.011	0.019	Bolting	853	13
27	7.0 W x 3.5 H	>21°–25°	0.010	0.011	0.016	Bolting	996	13
28	7.0 W x 4.0 H	>21°–25°	0.009	0.011	0.015	Bolting	1047	12
29	7.0 W x 4.5 H	>21°–25°	0.009	0.06	0.015	Bolting	1062	12
30	7.0 W x 5.0 H	>21°–25°	0.008	0.006	0.013	Bolting	1287	12
31	7.0 W x 5.5 H	>21°–25°	0.008	0.006	0.013	Bolting	1263	11
32	7.0 W x 6.0 H	>21°–25°	0007	0.006	0.013	Bolting	1280	10
41	7.0 W x 3.0 H	>25°–30°	0.011	0.011	0.021	Bolting	776	12
42	7.0 W x 3.5 H	>25°–30°	0.010	0.011	0.019	Bolting	847	12
43	7.0 W x 4.0 H	>25°–30°	0.009	0.011	0.017	Bolting	928	11
44	7.0 W x 4.5 H	>25°–30°	0.010	0.006	0.020	Bolting	827	10
45	7.0 W x 5.0 H	>25°–30°	0.008	0006	0.015	Bolting	1061	10
46	7.0 W x 5.5 H	>25°–30°	0.008	0.006	0.015	Bolting	1108	10
47	7.0 W x 6.0 H	>25°–30°	0.008	0.006	0.014	Bolting	1136	10
50	7.0 W x 4.0 H	>30°–35°	0.009	0.011	0.020	Bolting	813	10
51	7.0 W x 6.0 H	>30°–35°	0.007	0.006	0.015	Bolting	1065	9

The secondary and production drift cross-sections (with controlled convergence room-and-pillar) provided by Cuprum were supplemented with cross-sections for 4.0 m and 5.0 m high drifts, so that consistent half-metre height increments could be used.

Changes in the cross-sectional area and perimeter were analysed with changes in dip. It was determined that drifts with the same drift height and dips <20° could be combined and represented by the average area and average perimeter within a few percentage points variance, which is within the required accuracy of the study.

Only 4.0 m and 6.0 m high drift sizes in the ≥30°–35° dip categories were in the mine design, so only these drift size productivities were calculated in this dip range.

The room-and-pillar production drifts for room-and-pillar production with non-convergence have inclined backs parallel with the dip and flat floors like the convergence production panels, except that the ribs are vertical instead of canted. These production cycle times are shown in Table 16.19.

Table 16.19 Production Cycle Times (Crews for Room-and-Pillar)

Drift Type	Dimensions (Flat back)	Details (Ore dip)	Face Drilling Performance (h/t)	LHD Mucking Performance (h/t)	Bolting Performance (h/t)	Critical Activity	Critical Activity Performance (t/d)	Critical Activity Performance (m/d)
52	7.0 W x 3.0 H	≤20°	0.010	0.011	0.009	LHD Mucking	1477	22
53	7.0 W x 3.5 H	≤20°	0.010	0.011	0.008	LHD Mucking	1508	21
54	7.0 W x 4.0 H	≤20°	0.009	0.011	0.007	LHD Mucking	1510	19
55	7.0 W x 5.0 H	≤20°	0.008	0.006	0.007	Face Drilling	1981	20
56	7.0 W x 5.5 H	≤20°	0.008	0.006	0.007	Face Drilling	2049	19
57	7.0 W x 3.5 H	>20°	0.010	0.011	0.018	Bolting	883	13
58	7.0 W x 4.5 H	>20°	0008	0.006	0.015	Bolting	1082	12
59	7.0 W x 5.0 H	>20°	0.008	0.006	0.014	Bolting	1180	12
60	7.0 W x 5.5 H	>20°	0.008	0.006	0.014	Bolting	1194	11

Figure 16.25 Typical Panel Production

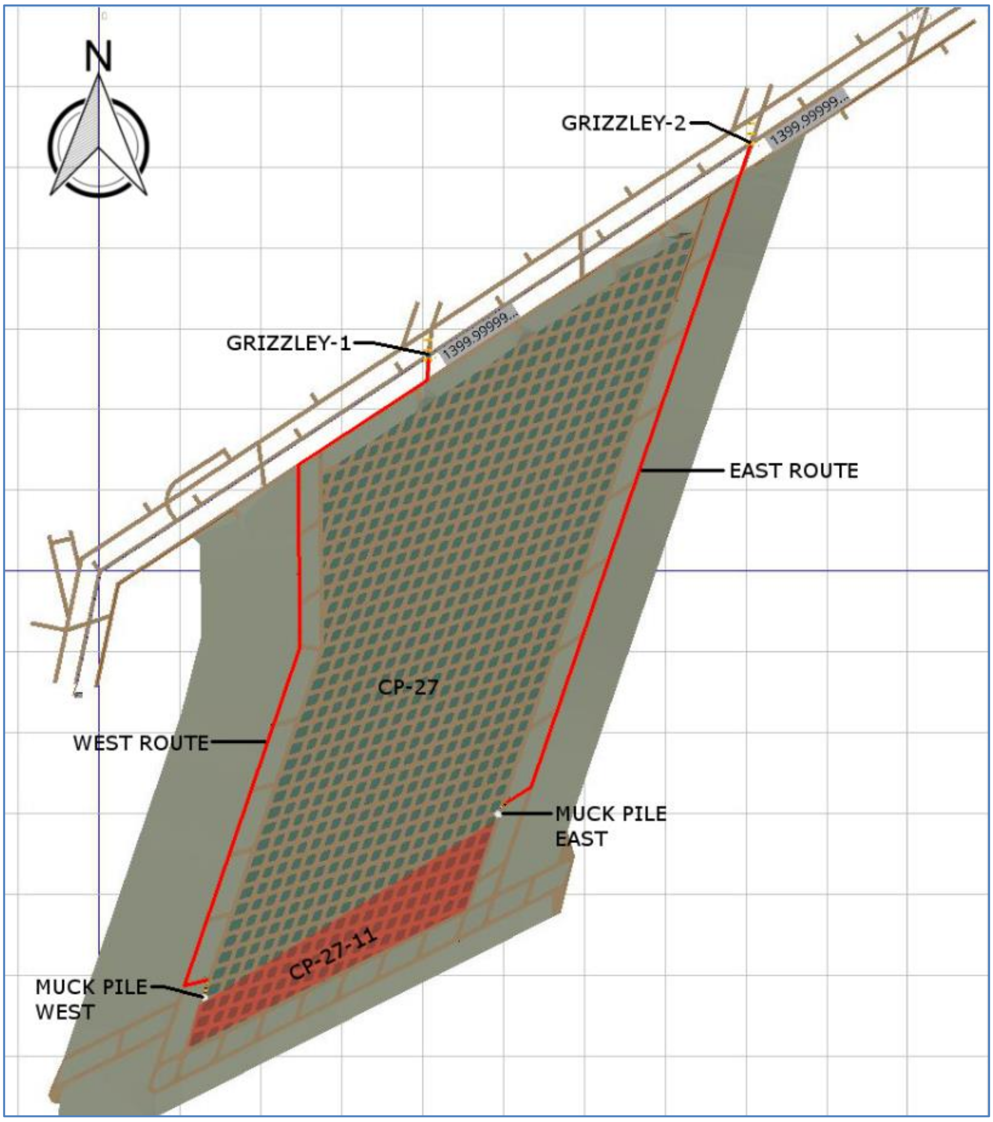


Figure by Stantec, 2017.

Pre-production Development Schedule

The initial development in the ramp-up periods will require a significant amount of the overall waste development that will be mined for the Kansoko Centrale and Sud mining areas. The waste development consists of the main infrastructure such as conveyor excavation, main shops and infrastructure, and dewatering settlers. Most ore development in this period consists of the service and conveyor declines, room-and-pillar mining, and the secondary development in preparation of panel production. Figure 16.25 illustrates the pre-production development schedule.

Figure 16.26 Pre-production Development Schedule

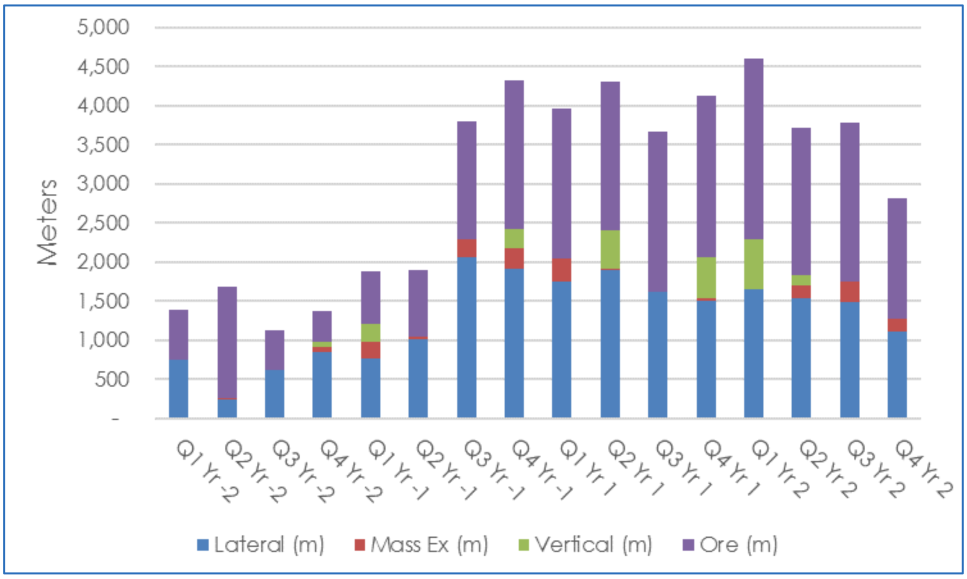


Figure by Stantec, 2017.

Life-of-Mine Development Schedule

The development schedule beyond the initial pre-production period targets the areas required to bring online the production panels that support the LOM plan. This would include excavating the primary and conveyor drifts ahead of production panels to access necessary ventilation raises. Figure 16.27 illustrates the LOM development schedule.

Figure 16.27 LOM Development Schedule

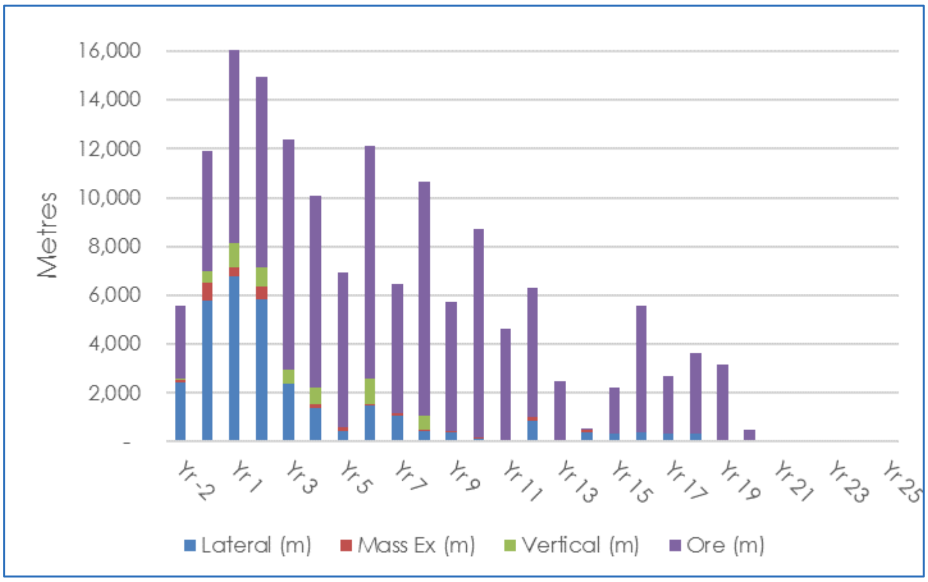


Figure by Stantec, 2017.

16.2.4.3 Mine Production Plan and Scheduling

Production Planning Criteria and Strategy

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

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

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

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

Production Ramp-Up Schedule

The following criteria were established for the targeted resource, to support the overall tonnage requirements from the Kamoa deposit. Table 16.20 details the targeted annual tonnages for the overall production requirements to meet the 6 Mtpa production rate.

Table 16.20 Production Schedule Criteria

Criteria	Details	
Initial and Ramp-Up	10.9 Mt	
Full Production	6 Mtpa	
Extraction / Recovery	75%	
Production Schedule	**Years**	**Tonnes**
Initial Production Mining (Year -2)	1	200,000
Ramp-Up (Year -1)	1	1,200,000
Ramp-Up (Year 1)	1	2,000,000
Ramp-Up (Year 2)	1	3,000,000
Ramp-Up (Year 3)	1	4,500,000
Full Production	17	102,000,000
Production (prior to ramp down)	22	112,900,000

Figure 16.28 illustrates the pre-production ramp-up and grades.

Figure 16.28 Pre-production Ramp-Up and NSR

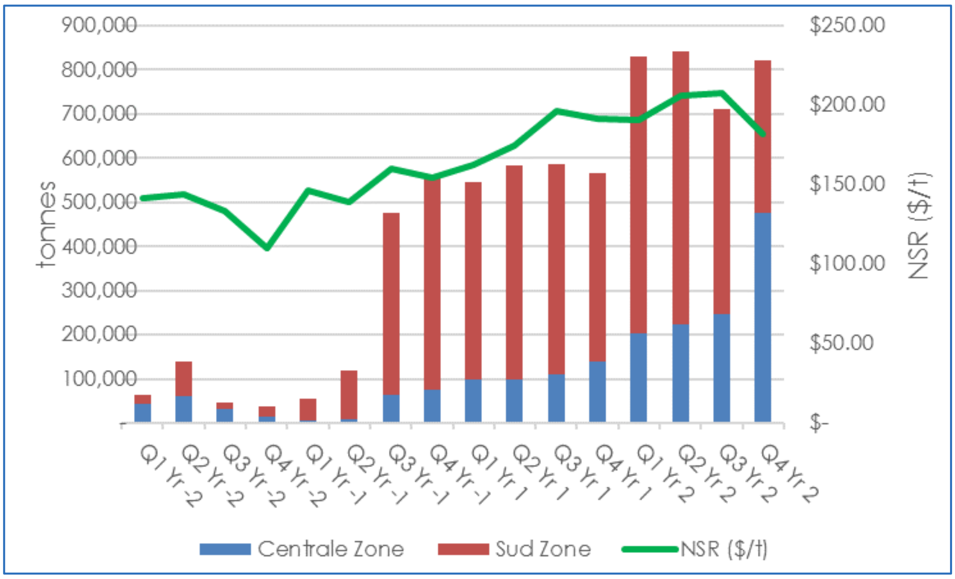


Figure by Stantec, 2017.

Life-of-Mine Production Schedule

Full production of 6 Mtpa is 17 years and tapers off as the resource is depleted. The panels were scheduled so that a higher NSR value is achieved earlier in the project. Figure 16.29 illustrates LOM schedule and grades. The mine production schedule is detailed in Table 16.21.

Figure 16.29 LOM Schedule and NSR

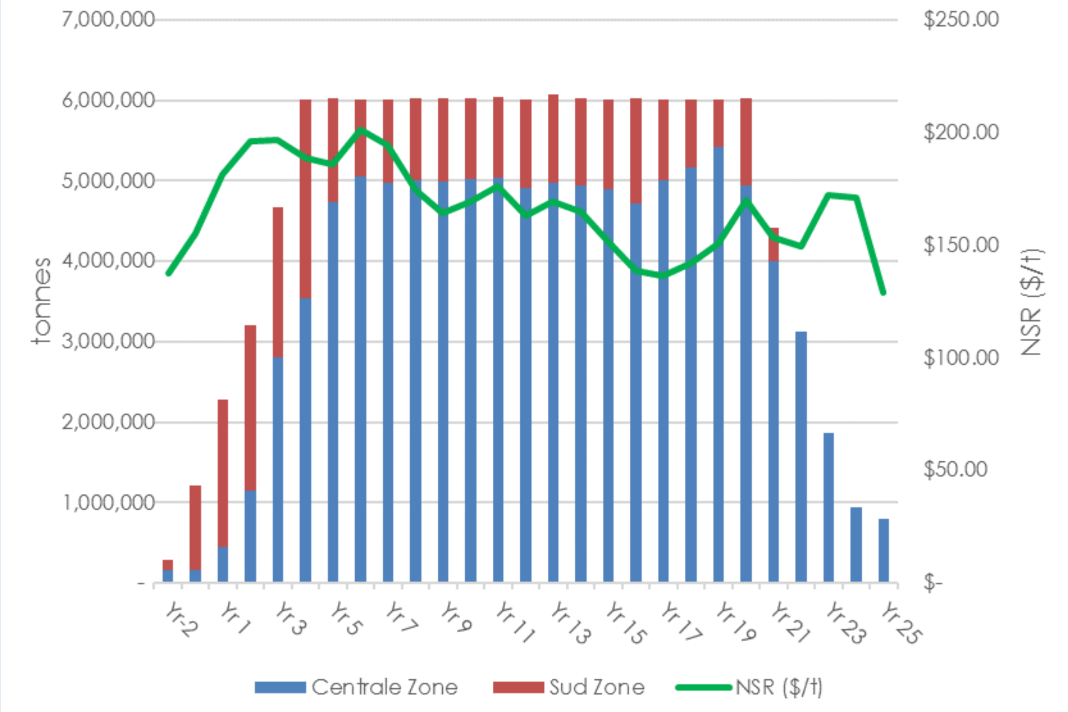


Figure by Stantec, 2017.

Table 16.21 Mine Production Schedule

Description	Unit	Total	Project Time (Years)													
			-2	-1	1	2	3	4	5	6	7	8	9	10	11	12
Room-and-Pillar Ore Mined	(kt)	3,397	-	-	347	656	1,033	446	640	275	-	-	-	-	-	-
	(% Cu)	5.29	-	-	5.49	6.12	4.82	6.28	4.92	4.09	-	-	-	-	-	-
Controlled Convergence Room-and-Pillar Ore Mined	(kt)	111,120	-	-	396	897	1,400	3,302	4,524	5,163	5,086	5,505	5,145	5,549	5,260	5,605
	(% Cu)	3.81	-	0.00	2.85	3.57	4.50	4.45	4.20	4.31	4.65	4.50	4.06	3.77	3.98	4.02
Ore Development Ore Mined	(kt)	10,665	-	289	464	729	772	919	850	589	923	510	882	478	767	445
	(% Cu)	3.34	-	3.15	2.63	2.83	3.68	3.44	4.06	3.37	4.00	3.20	3.42	3.55	2.97	3.65
Total Ore Mined	(kt)	125,182	-	289	1,206	2,282	3,205	4,667	6,014	6,028	6,010	6,015	6,027	6,027	6,027	6,050
	(% Cu)	3.81	-	3.15	3.52	4.07	4.40	4.42	4.26	4.21	4.55	4.39	3.97	3.75	3.85	3.99

Description	Unit	Total	Project Time (Years)													
			13	14	15	16	17	18	19	20	21	22	23	24	25	26
Room-and-Pillar Ore Mined	(kt)	3,396	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	(% Cu)	5.29	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Controlled Convergence Room-and-Pillar Ore Mined	(kt)	111,120	5,560	5,862	6,021	5,818	5,613	5,813	5,731	5,751	5,974	4,417	3,126	1,858	943	802
	(% Cu)	3.81	3.78	3.87	3.76	3.46	3.20	3.12	3.26	3.47	3.86	3.49	3.42	3.89	3.83	2.93
Ore Development Ore Mined	(kt)	10,665	448	215	1	187	406	198	278	267	48	-	-	-	-	-
	(% Cu)	3.34	2.80	3.29	1.93	3.44	2.79	3.01	2.82	2.83	3.54	-	-	-	-	-
Total Ore Mined	**(kt)**	125,182	6,007	6,077	6,022	6,005	6,019	6,012	6,009	6,018	6,022	4,417	3,126	1,858	943	802
	(% Cu)	3.81	3.71	3.85	3.76	3.46	3.17	3.12	3.24	3.44	3.86	3.49	3.42	3.89	3.83	2.93

Totals may not add up due to rounding.

16.2.5 Underground Infrastructure

16.2.5.1 Ventilation

The purpose of the mine ventilation system will be to provide sufficient air quantity and quality, dilute and remove air-polluting contaminants, control the thermal conditions of underground openings, and provide acceptable breathable air for working areas in the underground mine.

Diesel particle matter is the main driver for establishing airflow requirements for the underground openings as the baseline. These requirements will be later adjusted for the required cooling and refrigeration, increasing the cooling capacity of the ventilation system. To do this, initially the mine development and production schedule, in conjunction with underground equipment, will be considered to determine the required air quantity and primary flow distributions. Heat load calculations along with computer simulations with the variables obtained from the mine design will determine mine air cooling and refrigeration requirements.

The strategy will be to ventilate the mining sections with flow-through ventilation and avoidance of recirculation/reuse of air. Main service and conveyor declines will provide fresh air, while a ventilation raise near the bottom of the main declines (Vent Raise No. 1) will be used as an exhaust column. Fresh air from the declines will split into two major development fronts — one supporting Centrale and the other Sud. Bulkheads, ventilation curtains, seals, pillar sections, and booster fans will be used to control the air distribution within the panels.

Assumptions and Design Criteria

Assumptions and design criteria for the ventilation system are detailed in Table 16.22. The mine layout schematic illustrating the location of the surface ventilation raises is shown in Figure 16.30. The ventilation system is designed to provide localised fresh air intake for the major mining areas—Sud, Centrale North, and Centrale South—with dedicated exhaust assigned for each of the mining areas. The fans are located at the exhaust shafts on surface where possible, to reduce heat gain in the fresh air supply.

Table 16.22 Ventilation and Cooling Design Criteria

Item	Design Value
Surface Raises	4.0 m Ø and 5.5 m Ø
Internal Raise	3.0 m Ø
Drift Dimension	6.0 m x 5.5 m
Sud Conveyor Drift Dimension	6.0 m x 7.0 m
Steel Duct	1,525 mm Ø (60 inches)
Average Drift Friction Factor	0.0126 kg/m^3
Ventilation Raise Friction Factor	0.01 kg/m^3
Steel Duct Friction Factor	0.0037 kg/m^3
Airflow Requirements for Diesel-Powered Units	0.063 $m^3/s/kW$
Maximum Velocity in Drifts	6.5 m/s
Maximum Velocity in Ventilation Raises	24 m/s
Surface Elevation Mean Sea Level	1,436 m
Fan Location	Surface Except for Vent Raise No. 1
Assumed Leakage	15%
Surface Noise Level Maximum	45 dB
Fan Station Width	2 × Fan Diameter
Fan Station Length before Fans	5 × Fan Diameter
Fan Station Length after Fans	5 × Fan Diameter

Figure 16.30 Surface Ventilation Raise Layout and Dimensions

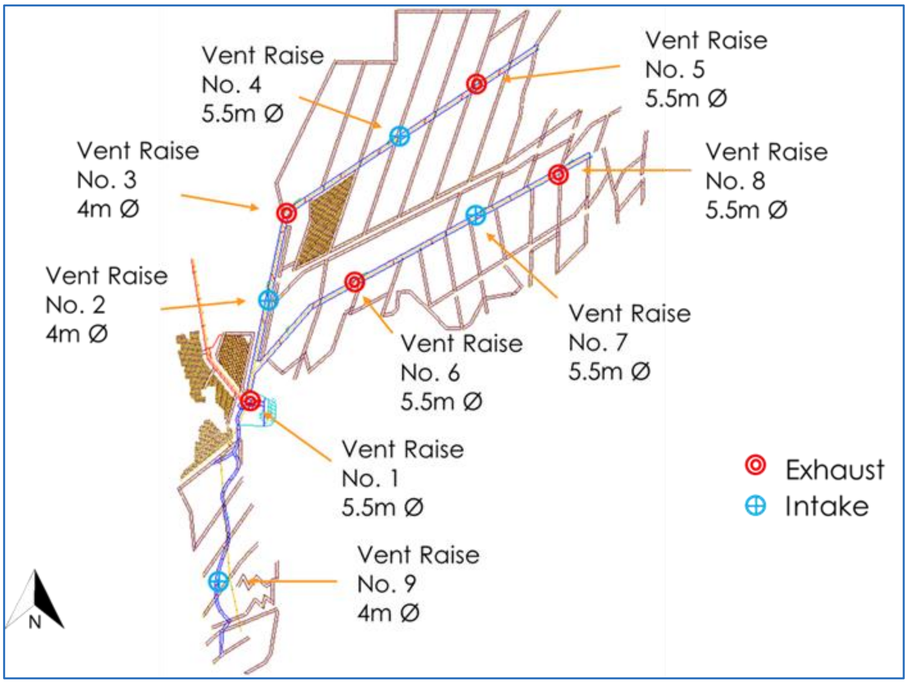


Figure by Stantec, 2017.

Mine Ventilation Philosophy

Planned production is set at 6 Mtpa, with comparable equipment to mine at this rate. Diesel exhaust gas dilution along with heat loads are the overriding factors.

The operating equipment fleet and locations will be assessed to determine the ultimate flow requirements and distribution. Mine ventilation and cooling will be developed through the main access declines and a series of ventilation raises, and primary development as the main ventilation arteries. Ventilation development will progress as mining requirements demand.

During main access decline development, the main service and main conveyor declines will be driven blind to every other cross-cut, where the cross-cut will be used to establish a loop for the next blind segment. When the loop is created, the service decline will be used as the return airways (RAW) and the conveyor decline will be the intake air. During the development of the blind headings, an exhaust overlap system will be used for maximum performance and safety.

The initial exhaust ventilation raise (Vent Raise No. 1) located at the bottom of the main declines will provide required ventilation to Centrale and Sud primary development headings. The main twin declines will provide fresh air, and the return flow will be exhausted through the ventilation raise.

As the mine life progresses, additional intake and exhaust ventilation raises will be developed to meet demand. All major ventilation fans, except those venting Vent Raise No. 1, will be installed at surface.

The primary development's "triple-line" to the mining areas will deliver fresh air as a flow-through system. Two primary service drifts and one primary conveyor drift will be driven blind to every other cross-cut, where the cross-cut will be used to establish a loop for the next blind segment. Each of the mining areas is designed to have a dedicated fresh intake shaft servicing Sud, Centrale North, and Centrale South.

Secondary Ventilation Systems

The following subsections describe methods for secondary ventilation.

Panel Development Headings

The secondary drifts define the panel and will be the primary ventilation route for the panel. These will be developed following the contour elevation of the ore, orientated close to the strike. These will be twin headings with cross-cuts between the drifts. During the development of the secondaries, one heading will be the intake and the other will function as an RAW.

Controlled Convergence Room-and-Pillar Panel Mining

Typical mining direction within the panel will begin at the extremities. Rooms and belts will be mined adjacent to the secondary at the extremities of the panel to establish flow-through ventilation. The typical ventilation circuit will flow through the secondary drifts into the active mining area once the connection is established. Fresh air will flush over the mining face due to the negative pressure from the exhaust side connected to an exhaust raise.

Air distribution within the rooms will be controlled with ventilation seals, curtains, pillar sections, and jet fans including booster fans.

Cross-cuts not requiring future access may be sealed with shotcrete walls. Primary fresh and exhaust airways are considered long-term development and will require corresponding long-term ground support.

Ventilation Controls

A combination of regulators and air doors, along with auxiliary booster fans, will direct airflow to the active mining areas. The possibility of using ventilation on demand should also be explored.

Airflow Requirements

Airflow requirements for the different underground mining crews are detailed in Table 16.23. The airflow required takes into consideration the utilisation factor of the mobile equipment and is rated at 0.063 m^3/s per brake horsepower, with utilisation factors applied. The equipment shows the crew requirement for development, production (room-and-pillar and controlled convergence room-and-pillar), and haulage of rock (ore and/or waste). The leakage throughout the mine was taken to be 15%, requiring a total flow of approximately 1,424 m^3/s at full production.

Table 16.23 Airflow Requirements for the Various Mining Crews

Item	Total Units (ea)	Engine Utilisation (%)	Unit HP (hp/unit)	Unit Vent ($m^3/s/unit$)	Unit kW (kW/unit)	Total Vent (m^3/s)
Development Crews						
LHD – Development	1	100	450	21	336	21
Bolter	1	15	78	1	58	1
Jumbo – Double Boom	1	15	185	1	138	1
Room-and-Pillar Crews						
LHD – Production	1	100	450	21	336	21
Prop Handler	1	50	185	4	138	4
Bolter	1	15	78	1	58	1
Jumbo	1	15	185	1	138	1
Controlled Convergence Room-and-Pillar Panel Crews						
LHD – Production	1	100	450	21	336	21
LHD – Loader	1	100	450	21	336	21
Bolter	2	15	78	1	58	1
Jumbo – Double Boom	2	15	185	1	138	3
Pillar Scraper	1	100	185	9	138	9
Truck	2	100	700	33	522	66
Haulage						
Haul Truck	1	100	700	33	522	33

This total flow requirement assumes that in the full production scenario, eight panels are active and four development headings are being driven, with five trucks to move material outside of panels. The summarised total airflow requirements for the various stages are summarised in Table 16.24.

Table 16.24 Airflow Requirements for Full Production

Description	Qty	%	Total Operating Equipment (kW)	Total Ventilation (m^3/s)
Subtotal Mobile Equipment				
Development Crews	4		1,460	92
Room-and-Pillar Crews	0		-	-
Controlled Convergence Room-and-Pillar Panel Crews	8		15,295	968
Haulage	5		2,610	165
Leakage		15		184
Total Ventilation Requirements				**1,410**
Subtotal Automotive Power			**19,365**	

16.2.5.2 Mine Air Cooling Facilities

Refrigeration will be required to provide sufficient cold air and to ensure that the development and panel reject temperatures remain within design parameters (i.e. average development and panel reject wet bulb temperatures of 28.5 °C). A first-order comparison of alternative refrigeration systems, notably underground refrigeration installation and surface ice makers, showed that the surface refrigeration using chilled water was the most economical. Ice systems are approximately 1.2 times the cost of normal refrigeration systems (underground melting dams, shaft pipes, pumping, remote heat exchangers, etc.), and underground refrigeration systems cannot be justified from an efficiency and operational perspective.

The Bulk Air Cooling (BAC) system will be installed on surface. The BAC system will be a horizontal-type, counter-flow, three-stage heat exchanger. Fresh air will be forced into the BAC chamber by means of three 200 m^3/second force fans positioned in parallel on the intake side of the BAC (600 m^3/second total). The intake mean summer wet bulb temperature will be 20°C, and the design outlet air temperature will be 10°C saturated. The BAC will be a direct contact-type system that provides the maximum heat transfer efficiency required. Chilled water will be sprayed into the moving air within the BAC chamber by means of spray nozzles separated in equal spacing along the chilled water pipes. The water droplets will fall to the bottom of the chamber and be reticulated back to the refrigeration plant evaporator plate heat exchanger system where the system is repeated.

To avoid overland piping and interference with community infrastructure, Kamoa's preference is for independent surface cooling installations. The proposal therefore is to locate discrete refrigeration plant rooms and heat rejection facilities at Vent Raise Nos. 7 and 4. The Vent Raise No. 7 plant will be sized for a nominal BAC duty of 10.0 MW; the plant at Vent Raise No. 4 will be sized for a nominal BAC duty of 4.0 MW.

BBE Consulting Canada's Kamoa Project – Mine Air Cooling and Refrigeration Facilities Evaluation and Prefeasibility Design report (BBE Consulting Canada, 2017).

16.2.5.3 Ore and Waste Handling Systems

Underground ore and waste handling will be designed for rubber-tyred and conveyor belt transportation of broken ore and associated waste, 360 d/yr. LHDs or haul trucks will transport the rock from the working headings. Ore will be moved to the surface via conveyor. Waste rock will be moved to surface using truck haulage, conveyor, or will be cast underground into the mined-out room-and-pillar areas.

16.2.5.4 Main Ore Handling System

The mine will be a mobile, trackless operation designed with a 6 Mtpa capacity for ore and waste handling. Bulk transport of ore and waste from the mining areas to the primary underground storage silos will be via a network of conveyor belts. LHDs and haul trucks will be used to transport ore from the mining panels, through tips, onto conveyors in the Centrale North, Centrale South, and Sud mining areas, respectively. Waste will be trucked out the mine directly to the waste dump on surface or into mined out areas underground.

The conveyors from Centrale North and Centrale South will converge onto a single conveyor that feeds an underground silo. Similarly, the conveyor belts from the Sud area will feed a second underground silo.

Ore will be fed through the silo onto a transfer belt, which will feed the main decline belts transporting the ore to surface. The conveyor head pulley of the final belt in the decline system will discharge into a splitter discharge chute for transfer of materials onto the process plant feed conveyor or onto a shuttle conveyor, distributing the rock onto one of two ground stockpiles—a waste stockpile and an ore stockpile.

Ore Passes

There will be two ore passes located in the Sud zone where the conveyor drift is located beneath the orebody. They will be equipped with fixed hydraulic rock breakers and sizing grizzlies (panel grizzlies) nested at the top of the ore pass. Mechanical feeders will transfer material at a controlled flow rate onto the conveyor.

Bins and Transfer Points

Production from the Sud and Centrale mining areas will feed into two vertical storage silos positioned at the bottom of the main decline conveyor (No. 1). The silos will control and regulate the feed onto the main decline conveyor belts and will also provide storage capacity to allow for maintenance of the belt and/or failure of one of the conveyor belts.

Each silo will have a dedicated feed from the Sud and Centrale mining areas, respectively. During the initial development phase of works, a haul truck tip arrangement will be available at the top of the silos, complete with a grizzly and rock breaker. This arrangement facilitates loading of the initial development rock into the silos, until the conveyor belts are installed and commissioned. Figure 16.31 illustrates a layout of the silos and the transfer belt.

Figure 16.31 Silo System Layout

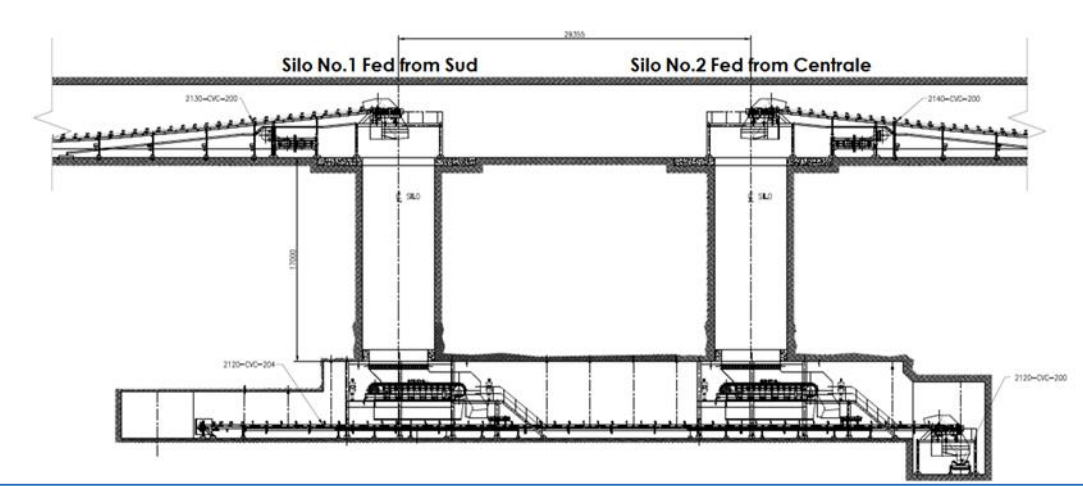


Figure by Stantec, 2017.

Silo Description

Both silos are 13 m high x 6 m in diameter, with a live capacity of 750 tonnes.

The silos will be lined so that self-mining after prolonged use does not occur. The silos will be lined with 40 MPa concrete and then painted with a sodium silicate cover, which will give the lining a final finished hardness of approximately 65 MPa.

The top of the silo infrastructure consists of the top of the silo steel structure, which supports the conveyors and flooring, and the top of silo the civil foundations and collar, which support the steelwork. The top of silo steelwork will provide the following:

- Support the Feed Conveyor Head Chute Loads.
- Silo Coverage.
- Maintenance Access.

The bottom of the silo includes all the infrastructure and equipment contained in the respective bulkhead including the throat concrete and chute, spiling bar system, bulkhead steelwork, chute work, apron feeders, and conveyor belt feed chutes. The silos will each have a single 1,500 mm x 1,500 mm outlet, sized four times the lump size so that bulk flow of material is attained. The chutes in system have been designed at an angle greater than 70° to ensure the material does not get hung up. The rock will flow through the silo onto an apron feeder with a designed capacity of 2,000 t/h, which will feed the rock onto the transfer belt. Figure 16.32 illustrates the side view of one of the silo bulkheads.

Figure 16.32 Silo Bulkhead Layout

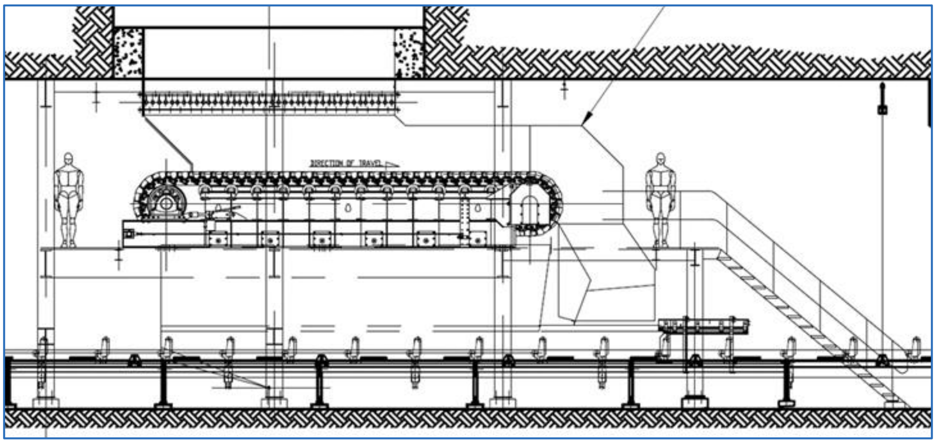


Figure by Stantec, 2017.

Grizzlies

There are grizzlies situated at the top of each of the ore passes, which will be identical throughout the ore handling system. Figure 16.33 is an indication of the grizzly configuration. The grizzlies incorporate the following features:

- The top of the ore passes will have grizzlies sized at 300 mm x 300 mm gauge grill.
- Hydraulic rock breakers will be installed at the top of each ore pass to handle oversized rock.
- Grizzlies will be constructed with a 50° slope to encourage self-scalping.
- Stop blocks will be constructed on either side of the grizzly to ensure that vehicles moving in the area do not drive in to the grizzly.

Tramp iron will be collected by hand.

Figure 16.33 Grizzly Layout

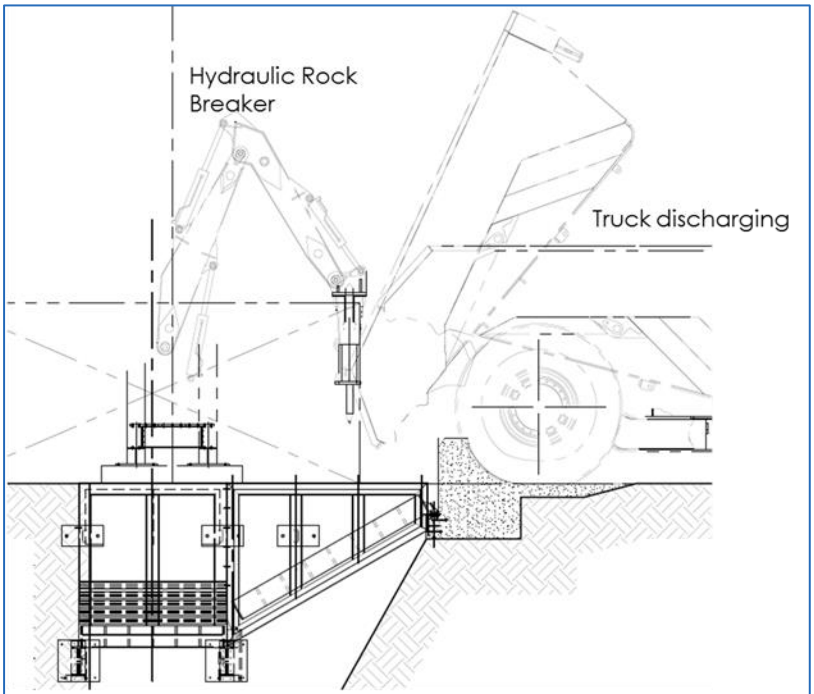


Figure by Stantec, 2017.

Conveyors

The primary bulk handling system comprises a network of conveyor belts in the following locations:

- Main Decline.
- Sud Conveyor Drift.
- Centrale North Conveyor Drift.
- Centrale South Conveyor Drift.
- Centrale Conveyor Drift.

Figure 16.34 indicates an overall layout of the conveyor belt network.

Figure 16.34 Conveyor Network System

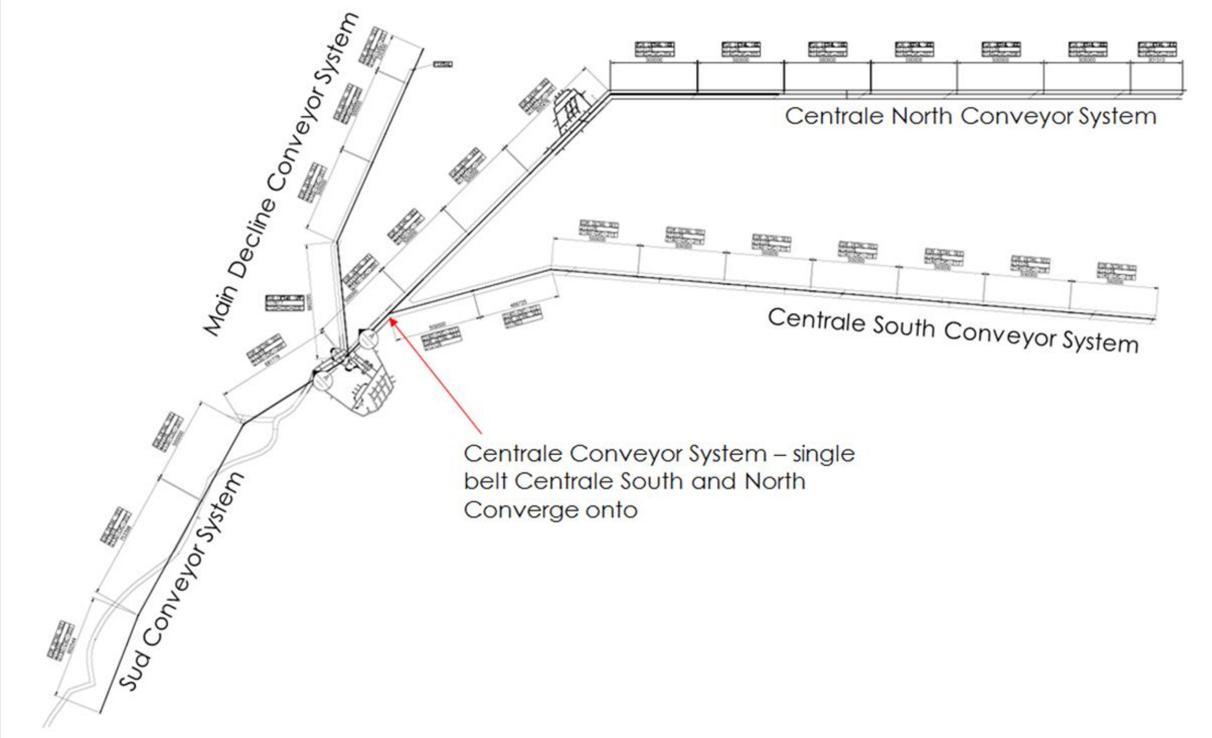


Figure by Stantec, 2017.

Main Decline Belts

The main decline conveyor system consists of four decline conveyors belts and a transfer belt below the silos. The flow of ore in this system is from the silo, onto a transfer belt, and then onto the four decline belts in series.

All the decline conveyors are designed to convey the total mine ore production of 6 Mtpa. Thus, all conveyors will have a capacity of 1,875 t/h and a belt speed not exceeding 2.5 m/s.

Sud Conveyor System

Ore will be transported from the face to two ore passes by dump trucks and LHDs. The ore passes will feed down to the Sud decline conveyor. Ore is also loaded with an LHD through a grizzly onto one of three 15 m long x 1,500 mm wide Class 2000 sacrificial belts. The belts support the two on-reef conveyor loading points within the Sud area, which feed the main Sud conveyor belt. The belt width standardises belting and components, and helps prevent spillage during loading.

The Sud conveyor system comprises four conveyor belts in series. All Sud conveyors are designed to convey ore at a rate of 6,000 t/d (mined out of two panels). All Sud conveyors will have a capacity of 488 t/h and a belt speed not exceeding 1.6 m/s. Conveyor Nos. 1, 2, 3, and 4, operating in a series arrangement, will transfer material from the ore passes and on-reef conveyor loading points into the Sud underground silo.

Centrale North and South Conveyor Systems

Ore will be transported from the face to the ore passes by haul trucks and LHDs. The ore passes will feed down to either the Centrale North or Centrale South decline conveyors.

The Centrale North conveyor system comprises 10 conveyor belts in series, and the Centrale South conveyor system comprises nine conveyor belts in series. All conveyors are designed to convey 12,000 t/d (mined out of four panels) ore production in Centrale North and Centrale South, respectively, with 900 t/h capacities and belt speeds not exceeding 1.6 m/s.

The ore and waste from Centrale North and Centrale South will then converge onto an 1,800 t/h capacity conveyor belt that feeds the underground silo.

Chutes / Feeders

Each of the ore passes will have a bulkhead containing a feed arrangement that feeds the respective conveyor belt.

Haul trucks or LHDs will discharge their loads through the grizzly and into the ore pass. A vibrating feeder will feed the rock onto the conveyor belt.

In-series loading will occur due to the layout of the ore passes, which is problematic for the following reasons:

- The conveyor feed chutes cannot extend to the level of the conveyor belt; otherwise, the chute will impede rock coming in from the back.
- If the feeders operate at full capacity and two or more ore passes in series are operating, the belt could become overloaded.
- The erratic nature of discharging into the ore passes (due to truck loading) makes it difficult to properly synchronise the system so that ore passes in the back end do not feed onto the belt concurrently with ore passes in front.

To ensure that continuous feeding onto the belt is achieved and that trucks do not wait at the top of the ore passes to discharge their loads, the chute system has been designed as follows:

- The vibrating feeder will be adjusted to load out of an ore pass at different rates—full or half rate ore, at quarter rate of capacity, depending on the number of tips in use in that area at that time.
- The conveyor feed chute is a nominal 600 mm from the top of the belt, giving enough clearance for the rock in the back to move past the chutes in the front.

Due to the elevation difference between the belt and the conveyor feed chute, it is anticipated that the belt life will be reduced due to impact loads. This effect will be reduced by the following:

- Rock travelling underneath the feed chutes (from the ore passes upstream).
- Skirts that will be extended to the bottom of the feed chute at an angle. When rock is discharged, most the impact will be taken up by the skirt rather than the belt.

Figure 16.35 indicates a typical conveyor feeder and chute arrangement.

Figure 16.35 Typical Ore Pass Discharge System

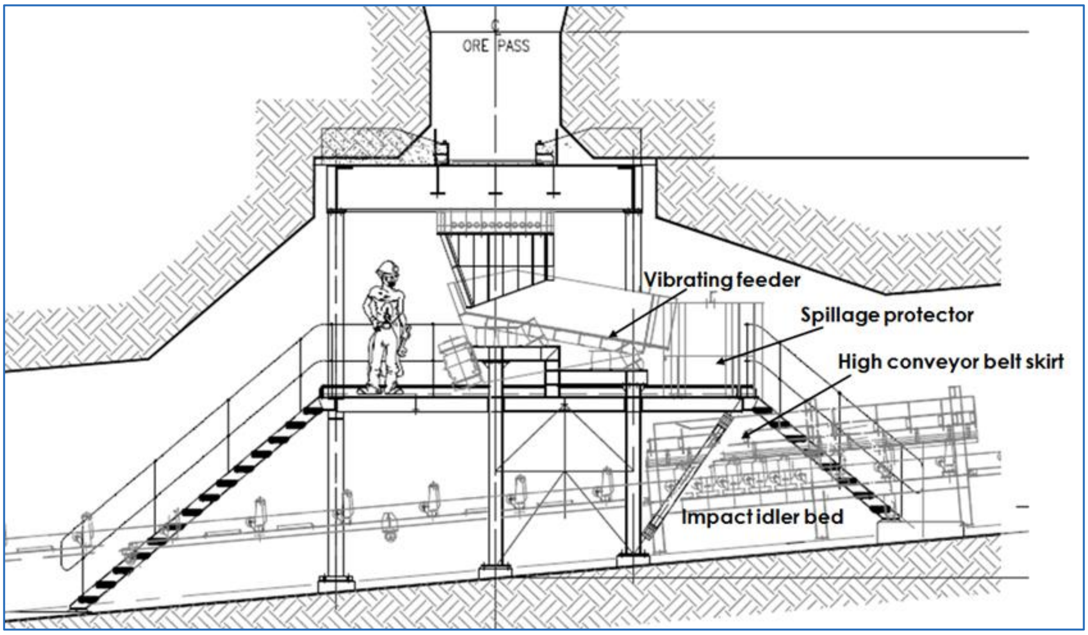


Figure by Stantec, 2017.

16.2.5.5 Workshops

Major mobile equipment will remain underground for the duration of the machine's life cycle and will be serviced/maintained in applicable underground workshops. Machines will only come out of the mine for a complete OEM refurbishment, or to be scrapped and replaced.

The final mine layout comprises two main workshops (main and Centrale) and several satellite workshops. The main underground workshop (see Figure 16.36) will be located near the intersection of the main decline and the primary accesses to the Sud and Centrale deposits. This underground workshop is central to both production mining areas. The Centrale workshop will be located near the bend of the northern decline at Centrale to reduce travelling distances from working places at Centrale.

As the mining progresses and travel distances increase, satellite workshops will be established near to production areas and furnished with the appropriate service equipment.

Production fleet vehicles operating mainly at the production face (drill rigs, bolters, and LHDs) will be serviced and maintained (minor repairs) at satellite workshops. All vehicles will revert to the main workshop for major services/repairs. Trucks hauling waste to surface as well as UVs will be maintained in the surface workshop. The surface workshop will also be equipped for rebuilds.

Figure 16.36 Main Workshop Layout

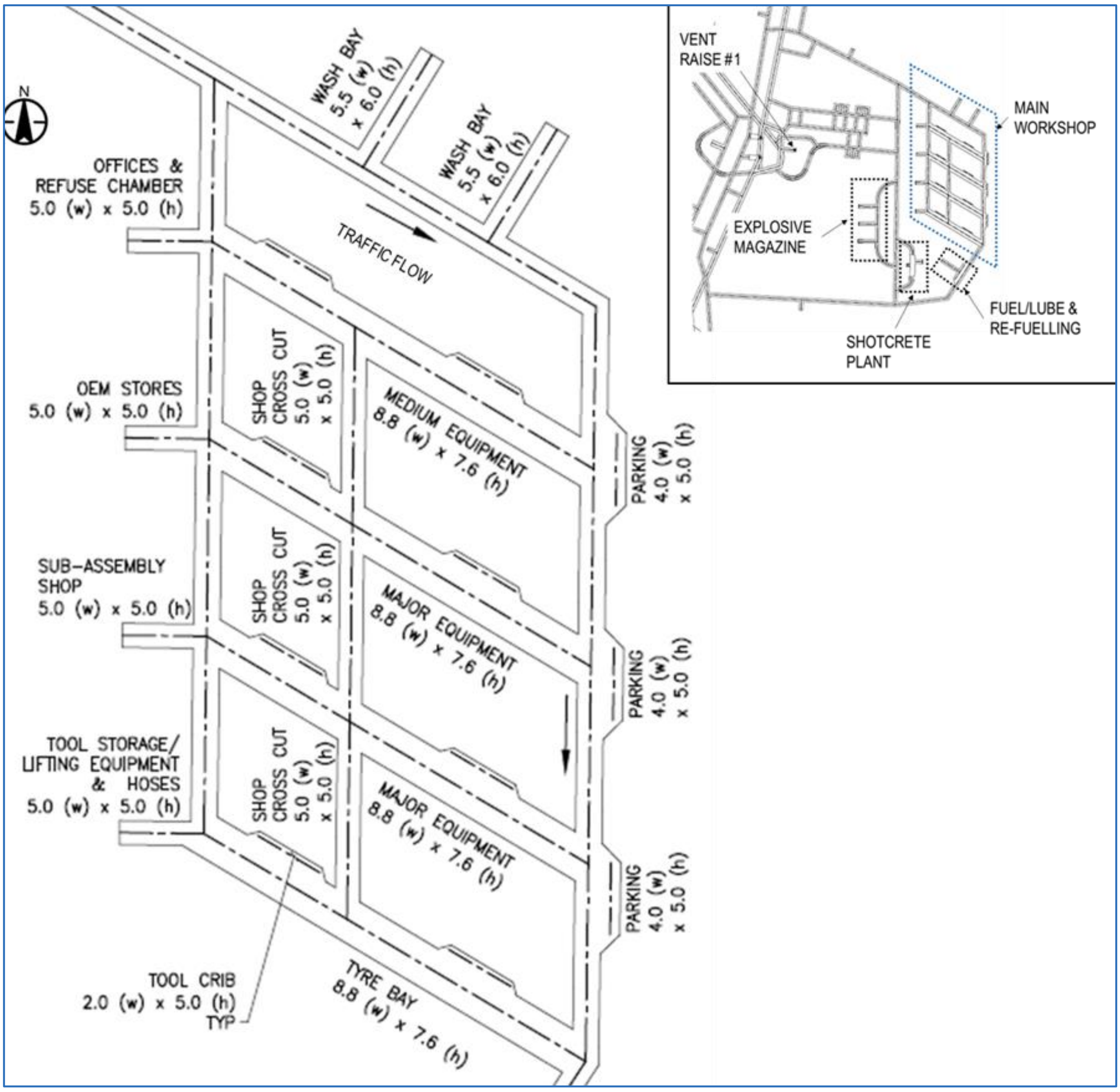


Figure by Stantec, 2017.

The Centrale workshop is illustrated in Figure 16.37 and comprises the same facilities as the main workshop except for having no tyre bay. Tyres will be transported from the main workshop when required.

Figure 16.37 Centrale Workshop Layout

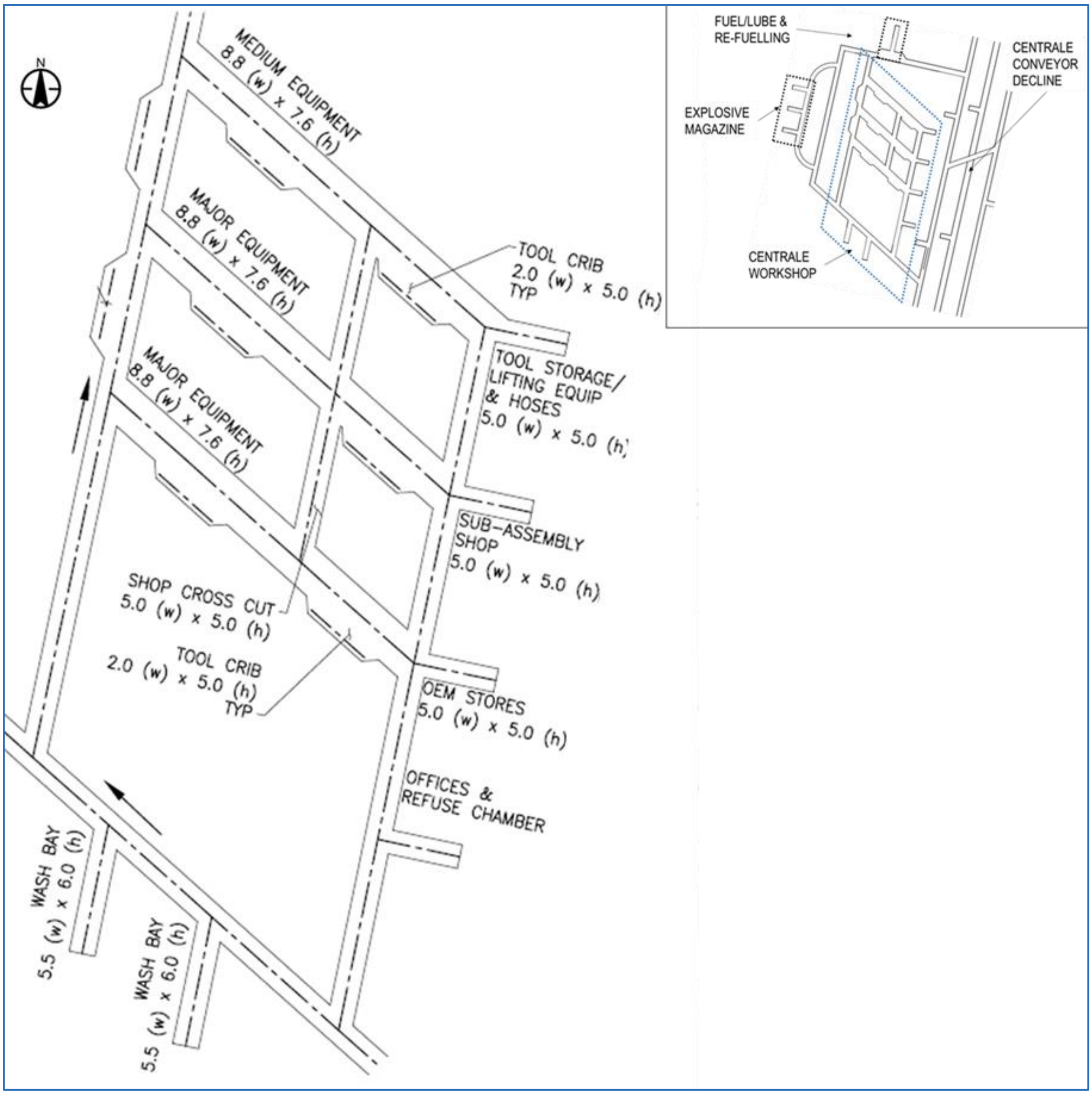


Figure by Stantec, 2017.

Large/Major Equipment Service and Repair Bays

Figure 16.38 shows the layout and sections of a large equipment service and repair bay, designed to aid towing of a broken vehicle onto the ramp, allowing ease of exit for the tow vehicle.

Figure 16.38 Major/Large Equipment Service and Repair Bay with Ramp

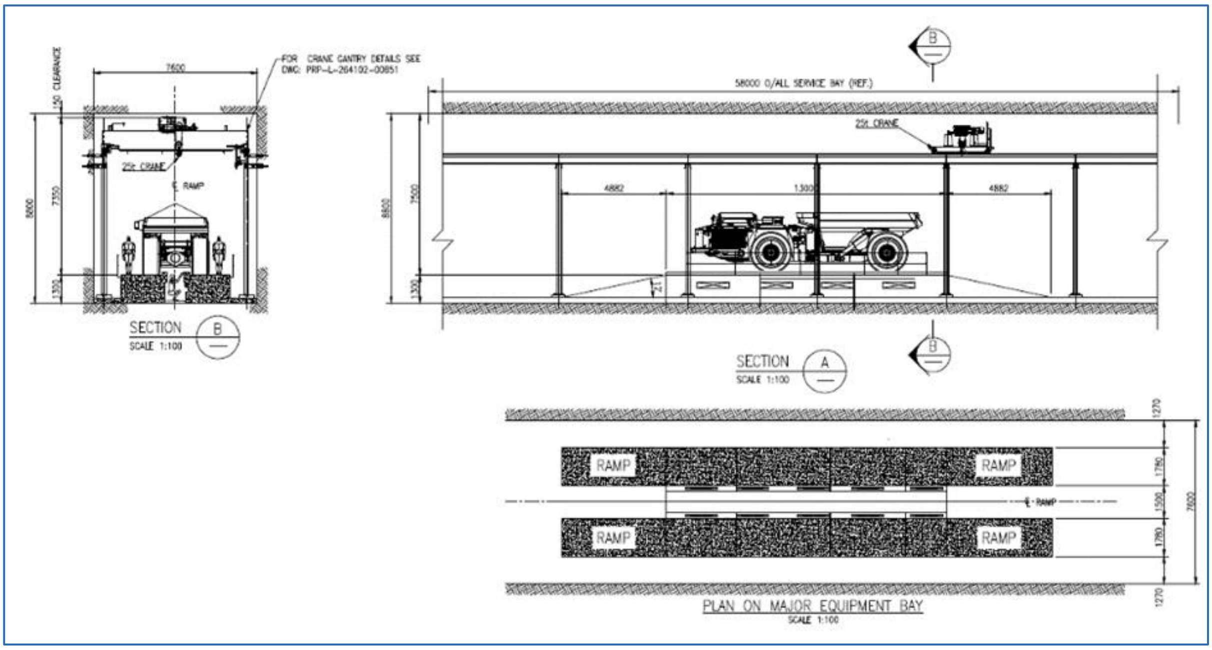


Figure by Stantec, 2017.

Medium Equipment Service and Repair Bays

The layout and operation of the medium equipment service and repair bays are similar to that of the major equipment (Figure 16.39). The dimensions are smaller, and the bays are equipped with a single cross-travelling 10-tonne overhead crane.

Figure 16.39 Medium Equipment Service and Repair Bays.

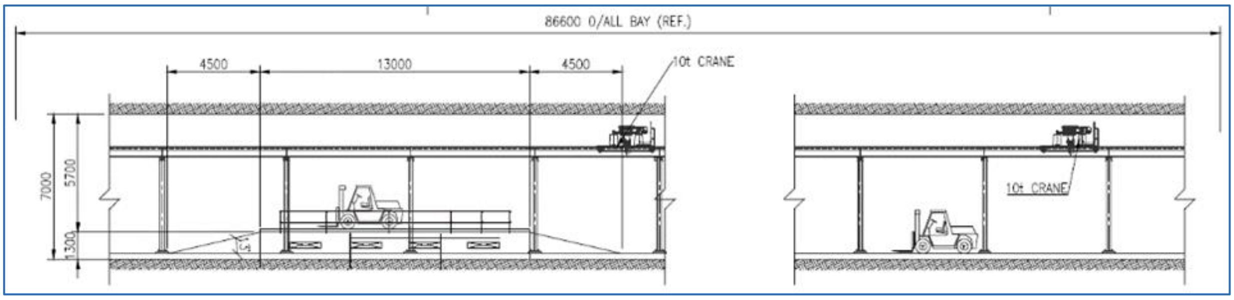


Figure by Stantec, 2017.

Tyre Bay

The bulk of tyres will be stored on surface. Minimum quantities of tyres will be stored underground in the allocated bay. Tyre repairs and fitting to the required rims will be undertaken on surface and transported to and from the workshops daily as required.

The tyre bay is equipped with a 5-tonne overhead crane. As shown in Figure 16.40, the bay has sufficient clearance around the vehicle to comfortably maneuver the wheel to the required position. Removing the mud guard of most vehicles allows the overhead crane to bring the wheel into the correct fitment position.

Figure 16.40 Tyre Bay

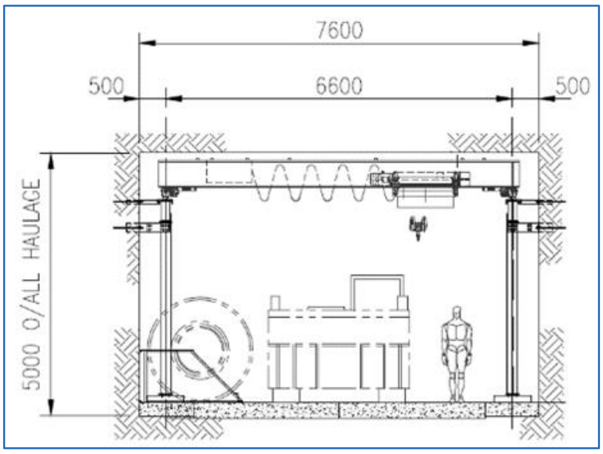


Figure by Stantec, 2017.

The multi-purpose vehicle (MPV) is equipped with a tyre handler to assist with changing wheels at the workplace or point of breakdown. The MPV will collect the tyre from the tyre bay and return the used/damaged tyre to the bay. The bay is equipped with racks for various rimmed tyres.

Satellite Workshops

As the extents of the mine increases, there will be a need for a limited number of satellite workshops equipped to perform daily maintenance on slow-moving mobile equipment (e.g. jumbos, bolters). The satellite workshops will have limited capacity and will potentially be single-bay. Concrete floors, lighting, and a hoisting arrangement will be incorporated in their design.

The satellite shops will be in proximity to the major ventilation raises in the area. Furthermore, they will have the fire protection necessary for such a facility. These facilities will not have access to compressed air via the main distribution system and must produce compressed air locally.

16.2.5.6 Fuel and Lubricant Distribution

The mine will operate as a trackless mining operation. The supply of fuel and lubricants is necessary for the operation of diesel-powered mobile and underground fixed mining equipment.

Fuel Distribution

A diesel and lubrication storage and distribution facility with refueling pumps will be constructed on surface, near the portal. These will be used initially until the facilities at the main workshop have been completed. The initial surface installation will be used for LOM for the refueling of surface vehicles and secondary mining fleet, such as trucks and UVs. There are four tanks on surface with a capacity of 83 m³ each, giving a total storage of 332 m³.

Once the underground facilities are commissioned at the main workshop, diesel will be piped down, through a dedicated borehole, from surface. The pipe column to the main workshop fuel storage area will be an "energy dissipation" pipe to prevent high-flow velocities and pressure build-up. The diesel will be batch fed to the underground storage tanks at the main workshop. Fuel will be batch pumped from the tanks at the main workshop to the tanks at the Centrale workshop via a 50 mm diesel pipe. Refueling stations will be available at both the main and Centrale workshops.

Fuel will be distributed to working sections from the Centrale and main refueling stations via diesel bowser cassettes. These cassettes will be used to refuel slow-moving or captive equipment such as drill jumbos, bolters, pillar scrapers, and shotcrete spray units.

Tier 2 diesel engines will be used, as 15 ppm ultra-low sulfur diesel is unavailable; 50 ppm diesel fuel is currently available on site.

Lube Distribution

Per requirements, lubrication oil will be stored in bulk tanks on surface and dispensed to the surface and underground workshops.

Bulk lubes will be transferred to surface workshops via dedicated pipelines and transported underground in lubrication cassettes. The underground lubrication cassettes will have dedicated storage areas.

Surface storage tanks were designed with sufficient lube storage capacity to operate the mine for approximately 30 days (per grade of oil).

Collection and Disposal of Waste Oil and Lubricants

Waste oil and other fluids will be collected in designated cassettes. The waste oil will be discharged from the collection container into a bulk tank, which, when full, will be transported out of the mine.

16.2.5.7 Explosives Magazine

Consistent explosive supply and distribution is critical for underground mining; the type, delivery, and storage thereof requires special design considerations. The underground mine will use the two-component emulsion system, consisting of a base product and a sensitiser combined at the face.

Explosives are classified per their respective hazard classification type, as follows:

- Explosive 1.1B: Nonelectric detonators, electronic detonators, or any type of blasting cap.
- Explosive 1.1D: Watergel explosives, ANFO, Cast boosters, detonating cord.
- Oxidiser 5.1: Oxidisers and base emulsion.

Each type of explosive will be transported underground separately and via different methods. Class 1 explosives, which include Explosive 1.1B and 1.1D, will be transported underground using purpose-built explosive cassettes. Oxidiser and base emulsion will be piped down from surface into storage tanks underground; distribution into the mining areas will be via emulsion cassettes.

Class 1 Explosives

Initially, there will be a surface magazine for detonators or ancillary explosives used underground. As mining progresses and permanent underground storage for explosives becomes available, the surface detonator and ancillary explosives magazines will be phased out.

Delivery of Class 1 explosives is assumed to be twice per week or per discussions with African Explosives Limited (AEL). Packaged explosives can be delivered to site within 72 hours' notice if required.

There will be two magazines located underground: main magazine and Centrale magazine. Each of the magazines will have three separate explosives bays—one for detonators, one for package explosives ("sticks"), and one bulk emulsion. The main magazine will be located near the main workshop complex at the bottom of the main declines. The Centrale magazine will be located near the Centrale workshop complex, adjacent to the entrance to the Centrale North and South mining areas. The recommendation for only two magazines is made to reduce unrestricted access by unauthorised personnel, to simplify inventory control and product issuance, and to improve the general security surrounding a limited number of explosives storage areas.

Each of the Class 1 explosives bays are designed with a storage capacity of 25 pallets and are of equal size and depth (5 m W x 25 m L), as illustrated in Figure 16.41.

Figure 16.41 Explosives Storage Underground – Layout (25 m L x 5 m W x 5 m H)

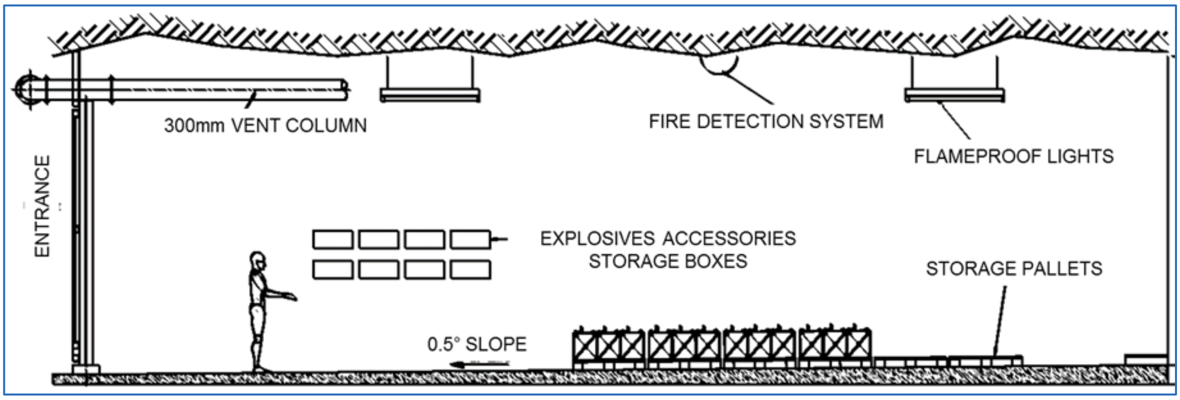


Figure by Stantec, 2017.

Bulk Emulsion and Sensitiser – Initial System

Initially, the emulsion and sensitiser will be stored in tanks on surface located near the portal. Emulsion trucks will deliver emulsion every second day; however, sufficient storage capacity for four days will be required (Figure 16.42).

Figure 16.42 Emulsion Initial Storage Facilities – Surface

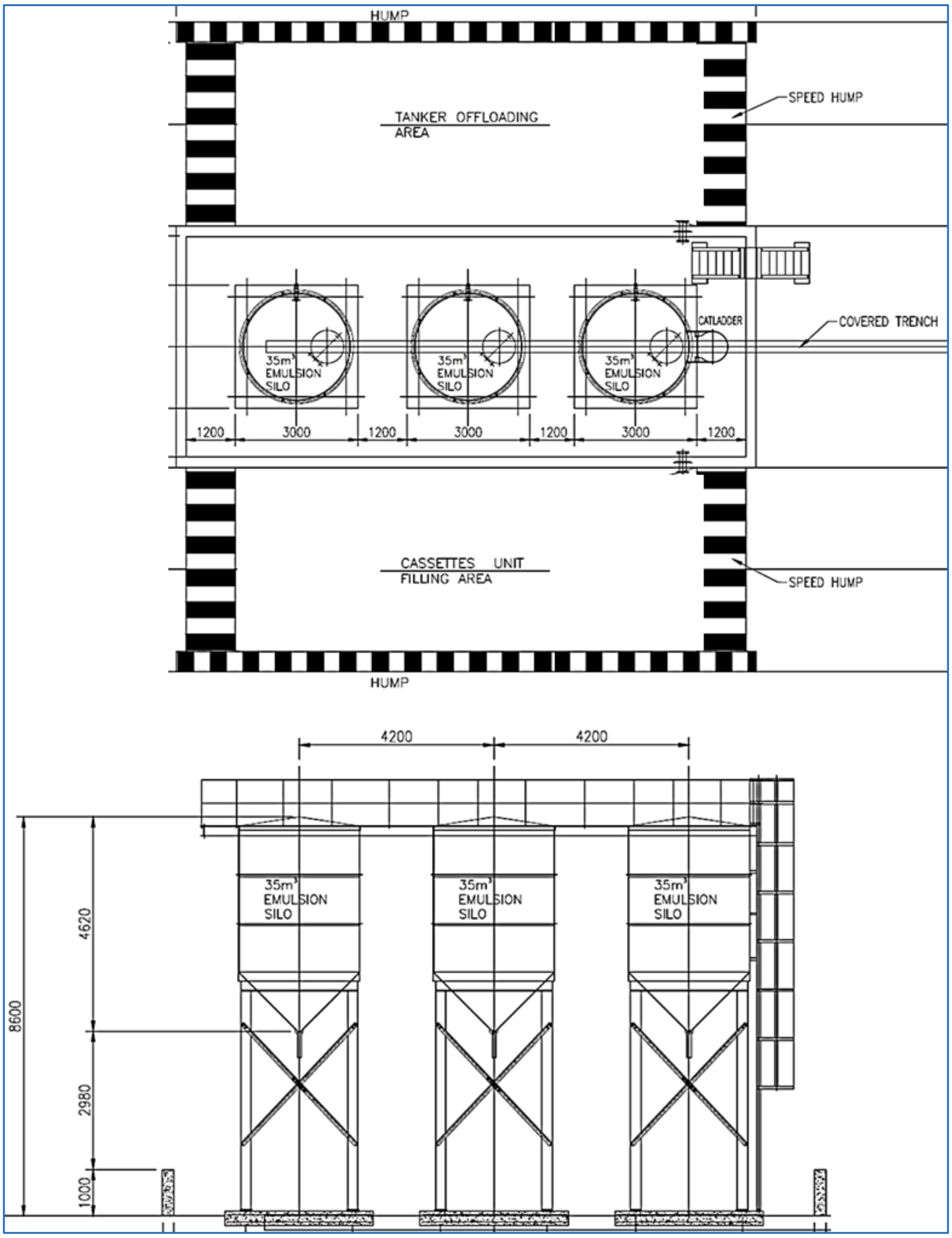


Figure by Stantec, 2017.

Bulk Emulsion and Sensitiser Vertical Drop – Permanent System

AEL was approached to obtain a cost for a suitable vertical drop system for Kansoko. The following criteria was used for the estimate:

- Total storage capacity underground – 90 tonnes.
- Storage facility underground, only.
- Vertical drop of 300 m.
- Bulk emulsion and sensitiser will be dropped, in separate HDPE pipes, through the same borehole.
- Emulsion and sensitiser will be transported using UVs to Centrale explosives storage areas and/or underground working areas.

The proposed system is well understood and in use at various operations in Zambia. Emulsion cassettes will be used to store explosives near working areas or in the Centrale emulsion storage magazine.

16.2.5.8 Concrete and Shotcrete Facility and Distribution

An LOM concrete and shotcrete batch plant will be built on the surface to deliver wet cementitious product. The product will be pumped from the surface batch plant to the borehole servicing the underground delivery facility. Prior to the borehole delivery system, a transmixer / agicar delivery via the decline portals will be used. A positive displacement pump connected to a pipeline will be used to transfer the cementitious mix from the batch plant to the borehole to underground.

The transmixer / agicar truck will take the shotcrete to the shotcrete placer pump truck and will discharge the load into the hopper of the shotcrete placement pump.

16.2.5.9 Compressed Air System

The compressed air system will not include a mine-wide reticulation system from the main surface facility. Piping will be provided from the compressor station on surface to the surface facilities, main underground workshop, and main refuge station. Permanent compressed air piping will be routed from surface through the decline to the underground workings.

Surface Compressor

Three 1,700 cfm compressors will be located on surface to supply air to the surface infrastructure and underground areas. The compressor will be connected to the mine's emergency power supply, ensuring that compressed air is supplied to the main refuge chamber during power outages.

Compressed Air Piping

Piping will be installed during the development of the primary declines and will extend into the underground maintenance workshop and into the permanent underground refuge chamber. General use compressed air will not extend to other sections of the mine; however, consideration would be given to piping extended to areas for localised use at facilities along the route if needed.

On-Board Compressors

On-board compressors will be available for utility work requiring compressed air. These will be sized for the equipment they will serve. Underground equipment/facilities that will use on-board compressors include the following:

- Jumbos (development, rock bolting, and cable bolting).
- Mechanic Service Trucks (e.g. lube, fuel, maintenance).
- Explosive Loading Trucks (to clean blast holes).
- Shotcrete Placing Equipment.

Electric Systems that Substitute for Compressed Air

Processes like valve actuation, which typically use compressed air, will be substituted with electrical actuators or hydraulic actuators. Underground equipment / facilities that will use this system include the following:

- Service Water Regulation Stations.
- Instrumentation and Controls.
- Chute Gates and Air Door Controls.
- Valves that Operate Remotely and/or Automatically.
- Ore Passes and Chute Cylinders (if required).
- Rock Breakers.
- Ventilation Doors.

16.2.5.10 Water Management

Water management from surface to underground will consist of managing service, potable, and fire water systems. The water management from underground to surface will consist of the mine dewatering system, which includes the production return water system and main pump stations.

16.2.5.11 Potable Water System

Potable water required for use by underground personnel will be provided from the potable water surface supply. The potable water storage will supply the surface facilities as well as the underground mine. A potable water pipeline will be routed from surface through the decline to the underground infrastructure. Pressure reducing stations will be installed as required. Potable water will be provided only for drinking and hygiene purposes and not for any other use.

16.2.5.12 Service Water System

Mine Service Water (MSW) is necessary for underground operations for drilling, muck pile wet down, wash bays, and dust control. An MSW pipeline will be routed from the surface through the decline to the underground, and to future development, and will remain for later use. Similarly, MSW pipelines will be installed in mine openings driven in ore for panel development and will be left for ore production needs. These pipelines will be removed from each panel for re-use after production is completed.

Pressure regulators will manage the increase in static water pressure created as the declines progress. Pressure-reducing stations will have redundant regulators for service and maintenance should issues occur.

16.2.5.13 Fire Protection System

Several fire suppression system types have been designed and catered for in the operating mine. In all instances, the systems comply strictly with the applicable codes of practice, both local and international. Each system has been designed as a fit-for-purpose solution, which protects the equipment and personnel without restricting operation. Table 16.25 illustrates the types of suppression systems used.

Table 16.25 Types of Suppression Systems

System	Typical Location
Medium Velocity Spray System	Lubrication rooms, lube packs, underground conveyors, and hydraulic power packs
High Velocity Spray System	Transformers
Free Agent Gas Suppression System	Substations and motor control centers
Foam / Water Deluge System	Fuel and lube storage on surface and underground
Hose Reels and Extinguishers	Site-wide, on all structures, in all buildings, both on surface and underground

Mobile Fleet Fire Protection

On-board foam-based fire suppression systems will be supplied and fitted by the mobile fleet OEM.

Fire Water Tanks

Fire water will be drawn from a dedicated source on surface and fed to the underground reticulation system, ensuring the availability of 675 m^3 at all times (size based on other projects with similar underground infrastructure layouts). The reservoir will be constructed with an internal division, subdividing the tank into two equal sections, thus guaranteeing at least 50% availability at all times during maintenance and/or mechanical damage. The tanks will be fitted with a dual suction. The tanks are designed to supply dedicated firewater for a duration of 90 minutes at maximum flow.

Fire Water Reticulation System

Fire water reticulation pipe work will be SANS 62 MED WT galvanised and banded pipe. All fittings and flanges will be Class 16. All isolation / section valves will be Underwriters Laboratories (UL) listed / Factory Mutual (FM) approved. The pipe will be installed in the conveyor declines and has been included in the conveyor designs and costs. The conveyors located in the main decline are installed in the intake and will be equipped with a fire line across the entire length of the belts.

Fire hydrants will be fed off the fire water column and will be placed no further than 60 m apart in the required areas.

Fire Detection in Substations / Motor Control Centers

All substations and MCCs will have both a smoke detection system in the room as well as a VESDA (Very Early Smoke Detection Apparatus) in the cabinets.

Each installation will have its own panel for remote status monitoring via potential free contacts. Each installation will be zoned accordingly, requiring a double knock (two adjacent zones) in simultaneous fire condition prior to the discharge of the gaseous suppression system, thus preventing the possibility of accidental discharge.

Each panel will also contain potential free contacts used for the shutting down of associated equipment (e.g. main incomer, air conditioning system).

Fire Detection on Conveyors

Flame detectors will be placed at strategic locations and will detect a moving fire in its incipient stage. The detection system will initiate belt shutdown and will activate the solenoid on the associated deluge valve. Each detection system will have its own control panel with potential free contacts for belt shutdown as well as remote monitoring of fire and fault signals.

16.2.5.14 Materials Handling Logistics

Materials, equipment, and mining supply items will be delivered by road to the mine site warehouse located at the surface. The mine site warehouse will manage and source services for both the process plant and mining operations.

The mine site warehouse will supply the underground maintenance facilities and mining laydown areas with the following:

- Ground support bolts, screen, utility service piping, pipe brackets, power cable, communication cable, ventilation duct, fans, etc.
- Spare parts for the mobile production fleet, which will be kept at and managed from warehouse facilities within the workshops.
- Tyres for the mobile production fleet, which will be requisitioned from the mine site warehouse and stored in the tyre workshop.

Mining and engineering materials and spares will be transported underground by rubber-tyred diesel-driven cassette carriers.

16.2.5.15 Underground Storage Areas

Designated underground storage areas will be located throughout the mine and typically in proximity to the point of use. Storage areas designated for infrastructure support (e.g. explosive magazines, fuel and lube, warehouse items in transit) will have permanent ground support including shotcrete. These areas will have concrete floors and lighting.

Mining supplies will be managed and sourced from the surface mine site warehouse and will be kept in laydown areas close to the mining operations.

Main Laydown

The main laydown area will be designed as a drive-thru. The area will store pipes, conveyor belts, power cable, fans, and other consumables. An excavation 10 m wide will be large enough to store large items and still leave room for vehicles to pull through. Assisted lift devices will be used to eliminate back injuries (e.g. pallet jacks, overhead cranes, mobile lift trucks).

16.2.5.16 Refuge Stations

Refuge stations will be required to house underground mining personnel in a secure, hazard-free location during emergency conditions. A constructed or modular-style refuge station will be located near the underground maintenance workshop area. If the workshop refuge station is not serviced with suitable compressed air from surface, it will be equipped with self-contained breathing systems. Portable refuge stations will be used to maintain compliance as the mining development faces advance. In the event of an emergency, a notification system, with backup, will signal all personnel to stop work and proceed to the nearest refuge station. All refuge stations will be sized to meet the capacity requirements for the area.

16.2.5.17 Toilet System

Underground sewerage will comprise two systems: fixed flushing toilets at the main workshops and mobile flushing (non-chemical) toilets for the remainder of underground workings.

The mobile toilets are designed as utility vehicle attachments and are easily maneuverable. Each unit will be fitted with a sump and pumped empty into a sewage tank mounted on a cassette carrier. This tank will be transported to surface and emptied into a sewerage disposal system on surface.

16.2.5.18 Power and Communication Systems

Electrical Substations and Power Distribution

Power will be distributed at 11 kV to the underground mine switchgear from two surface feeder breakers, for redundancy. The underground switchgear will be contained in an E-room and will have separate feeder breakers feeding major mine areas for isolation purposes and to minimise large connected loads to each feeder. Each feeder will feed multiple mine power centres, which will step down the voltage to 525 V for centralised operational loads.:

- Mine Medium-Voltage Distribution: 11 kV.
- Secondary Distribution: 525 V.
- Low-Voltage Distribution: 400/230 V.
- Frequency: 50 cycles per second.

The power will be spread to Centrale North, Centrale South, and Sud through the primary development headings. The power will feed the main fixed equipment, such as the conveyors, and the production panels.

The full power requirements for surface and underground are listed in Table 16.26. Surface loads consists of surface production fans and the cooling plants for the underground mine.

Table 16.26 Total Power Requirements

	Max Diversified Running (kW)	Max Peak (kW)	Max Peak kW 20% Contingency (kW)
Total UG	13,719	26,013	31,215
Total Surface	5,529	5,937	7,124
Total Surface and UG	19,248	31,950	38,340

16.2.5.19 Communication, Controls, and Automation Systems

Communications System

The backbone for the communications system is based on a redundant fiber network. This system will be used to support all voice and data communication requirements for the Project.

Radio communications for the mine will be provided over a leaky feeder system, which will be distributed throughout the entire mine for communication purposes, incorporating hand-held and fixed radios. This will be used to support the people detection system (PDS) / vehicle detection system (VDS), and ventilation-monitoring systems. The leaky feeder can also be used for central blasting.

Control System

The mining control will be located on surface in the main surface office, for control of daily mining operations on surface and underground. The equipment provided within these facilities is detailed in the control and instrumentation design criteria.

Cameras will be installed at each rock breaker, conveyor transfer point, and pump station. Fiber will be installed for monitoring of the power system and control for conveyors, pumps, and rock breakers. A fiber allowance has been made for ventilation-on-demand, if required.

Underground Access Control

Upon entering the mine site through the surface access controlled complex, mining personnel will proceed to the Change House and Lamp Room. Access into and out of the mine will be controlled by means of an electronic tag-in / tag-out system integrated into the cap lamps, which is monitored in the Control Room.

16.2.6 Mining Equipment

Criteria considered in equipment selection include suitability, equipment standardisation, and cost. The equipment selection process was iterative and aimed at obtaining the optimum equipment required to achieve the planned development and production quantities and rates.

The equipment requirements are split into two categories: fixed and mobile. The equipment requirements for each category are estimated at a prefeasibility level of accuracy and cover the major components for the operation.

All fixed and mobile equipment used for development and production activities will be based on a 6 Mtpa ore production and associated development. The schedule for equipment purchases and replacement will be based on a rebuild and replace cycle. No equipment will be replaced within 2 years of the end of the LOM.

16.2.6.1 Mobile Equipment

Primary

The average primary mobile equipment fleet is based on specific work activities per the mine schedule. Equipment types – high profile versus low profile – will vary based upon the areas mined in any given year.

Secondary

The secondary mobile equipment fleet is based on previous study experience for this and other projects, including the following underground mobile equipment:

- Light Vehicles (manpower).
- Utility Cassette Transports with Cassettes.
- Graders.
- Skid-Steer Cleanup LHD.
- Portable Welder Trailer.
- Concrete Pump Trailer.
- Explosives Loading Trucks.
- Shotcrete Sprayer.

Rebuild and Replace

The rebuild and replacement of equipment is calculated based on the life, during operating hours, of an individual piece of equipment. Equipment life is calculated using operating hours as well as vendor-provided actual operating hours for similar operations. Adjustments between engine (diesel) and electrical (percussion for drilling equipment) hours are segregated. Table 16.27 lists mobile equipment types with typical rebuild / replacement hours.

Table 16.27 Mobile Equipment Rebuild and Replacement Operating Hours

Description	Operating Hours Prior to Rebuild	Operating Hours Prior to Replacement
Jumbo – High Profile	15,000	25,000
Jumbo – Low Profile	15,000	25,000
Rock Bolter – High Profile	15,000	25,000
Rock Bolter – Low Profile	15,000	25,000
Cable Bolter	15,000	25,000
Haul Truck – 51 t	19,200	32,000
LHD – 9.8 t	15,000	25,000
LHD – 17 t	15,000	25,000
Emulsion Loader	15,000	25,000
Grader	12,000	20,000
MPV Transporter	12,000	20,000
Shotcrete Spray Machine	12,000	20,000
Pillar Scraper – High Profile	12,000	20,000
Pillar Scraper – Low Profile	12,000	20,000
Prop Handler	12,000	20,000
Light Vehicle	n/a	3 years

Quantities

The mobile equipment is listed in Table 16.28.

Table 16.28 Mobile Equipment List

Description	Yearly Max. Req.	Purchase/Replace	Rebuild
Jumbo – High Profile	14	14	0
Jumbo – Low Profile	10	10	0
Rock Bolter – High Profile	14	14	0
Rock Bolter – Low Profile	10	10	0
Cable Bolter	4	6	8
Haul Truck – 51 t	18	60	50
LHD – 9.8 t	13	26	25
LHD – 17 t	37	68	68
Explosives Truck	11	47	40
Shotcrete Sprayer	2	2	1
Rock Ripper – High Profile	5	19	15
Rock Ripper – Low Profile	7	20	16
Prop Handler	2	5	4
Grader	4	29	27
Utility Equipment UG	24	36	18
Light Vehicle	26	131	0

16.2.6.2 Fixed Equipment

Electrical Equipment

Electrical Switchgear

- The MV switchgear will be indoor, metal-enclosed, vacuum circuit-breaker type. The switchgear will contain withdraw-able circuit breakers and be fully tested in accordance with the applicable standards. The metal-enclosed switchgear, under both short-circuit and internal arc conditions, will be designed to prevent injury to people operating the switchgear.

Transformers

- Both power and distribution transformers for surface operation will be oil-insulated, double-wound, three-phase units with lockable, off-load tap-changing facilities. The transformers will be capable of operating continuously without adverse effects, including overheating, under all specified conditions of operating. The cooling method will be Oil Natural Air Natural (ONAN). The transformers will have a weatherproof control panel containing all auxiliary wiring.

Substations

- Dry-type transformers with cast, epoxy-resin encapsulated windings will be installed in all underground substations. The resin encapsulation will be moisture-free, non-hygroscopic, flame retardant, and self-extinguishing.

Mini-substations will provide power for the surface and underground small power and lighting loads. The mini-substations will be three-phase and fitted with oil-insulated, double-wound transformers complete with a ring main unit.

Major fixed equipment is defined and addressed within the construction items where they are used, based on the mechanical equipment list. Minor fixed equipment (e.g. drift fans, face pumps, safety equipment) is included as an individual line in the Owner's costs.

Table 16.29 summarises the main fixed equipment for the Kamoa mine design.

Table 16.29 Fixed Equipment

Description	Item Qty	Description	Item Qty
Materials Handling		**Electrical and Communications**	
Surface Transfer Tower	1	Main Substation	
Surface Shuttle Conveyor	1	MLCs	
Silo Mechanical	2	Leaky Feeder System	
Tips for Sacrificial Belts	2	**Safety and Miscellaneous**	
Conveyors	30	UG Safety Equipment	
Rock Breakers and Tips	29	Portable Refuge Chambers	
Ventilation		**Surface Facilities**	
Main Fans	10	Fuel and Lubrication Facility	1
Development Fan	87	Concrete / Shotcrete Facility	1
Development / Production Fan	44	Temporary Emulsion Storage Facility	1
GZRM Skid Production Fan	22	Permanent Emulsion Storage Facility	1
Air Doors – Pair	4	**Underground Facilities**	
Mine Air Cooling Facilities (4 MW, 10 MW)	2	Main Workshop Mechanical and Tools	1
Mine Service Water		Centrale Workshop Mechanical and Tools	1
Metso Pumps	7	Satellite Shop Jib Cranes / Fire Doors	3
Mine Dewatering		Emulsion Storage Facility	1
Portable (Sump) Pumps	72	Concrete / Shotcrete Facility	1
Metso Pump (250 kW)	15	Fuel and Lubrication Facility	1
Pump Skids	109		

16.2.7 Personnel

Personnel requirements were developed to support planned development, construction, and operation requirements for the mine. Only personnel directly linked to the operation of the mine are included in this study. Personnel that share other Project activities (e.g. accounting, training, personnel management, environmental, permitting, housing, security, ambulance) are excluded. Personnel requirements are not determined for the following factored personnel:

- Owner's Project Team.
- EPCM Team.

Figure 16.43 illustrates the average annual personnel requirements for the Project over the LOM.

Figure 16.43 Contractor Vs. Owner Personnel Summary

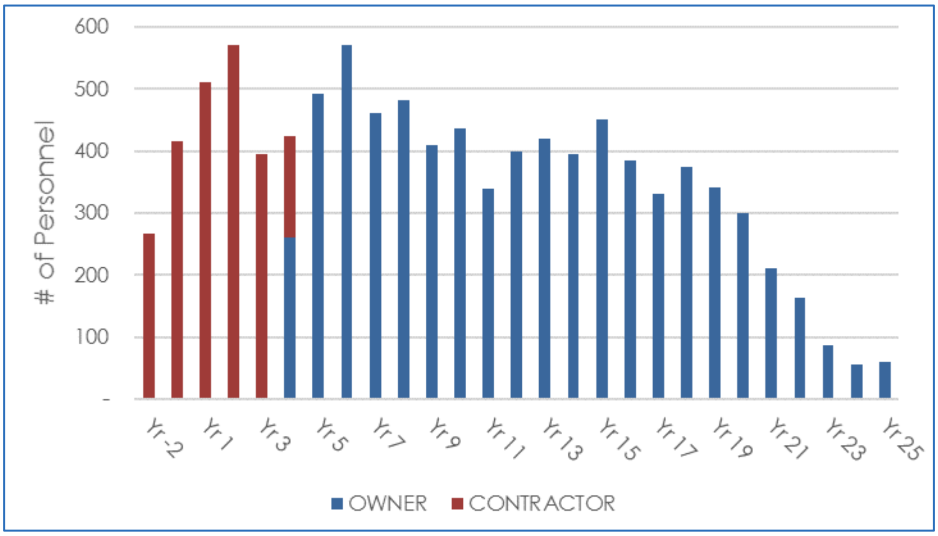


Figure by Stantec, 2017.

16.2.7.1 Training

Competent mining crews, in particular mobile production equipment operators, are essential in safely achieving production targets. A training department for both mining and engineering has been allowed for in the labour complement. A training facility will be available on surface for technical training. Practical training will be carried out underground, on the job, where final assessment for certification will be done. Recruitment of local labour will require training to be conducted in French and Swahili.

16.2.7.2 Criteria

Direct and indirect labor requirements were established to suit the selected mining method, support systems, and general mine requirements during mine development, construction, and operations. Personnel requirements are based on an operating schedule of 12 hours per shift and two shifts per day. Contractor crews will work 360 days per year. Owner capital work and production are accomplished in 360 days per year.

16.3 Open Pit Potential

Open pit mining was previously studied in the Kamoa 2013 PEA. This work was reviewed during the prefeasibility study. The results of the Kamoa 2013 PEA open pit analysis continue to be valid but have not been included in the mine planning for the Kamoa-Kakula Development Plan because the underground production schedule meets the plant capacity requirements.

The open pit resource represents an opportunity as a readily available alternative source of plant feed if delays were to be experienced in underground production or additional feed were required. The open pit resource will need to be brought into reserve category, further study of the relative ranking by value to determine the relative ranking of the open pit and underground resources could be undertaken to determine the timing requirements for their development.

17 RECOVERY METHODS

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

The Kamoa 2017 Development Plan is based on the flowsheet envisaged for the Kamoa 2016 PFS, updated to cater for increased throughput in the Kamoa 2017 PFS. Note that as Kakula is only at PEA and is not described in this section but in Section 24.

17.1 Introduction

This section on recovery methods incorporates assumptions, analysis and findings of the Kamoa 2016 PFS, updated to cater for increased throughput and copper grade.

The Kamoa 2016 PFS process plant consisted of a 3 Mtpa Run-of-Mine (ROM) concentrator incorporating staged crushing, ball mill grinding and flotation. The Kamoa 2017 PFS process plant is based on the same concentrator principle, but at a throughput of 6 Mtpa ROM. The output of the process plant is copper concentrate which is sold.

17.2 Process Description

Ore will pass through a 300 mm square grizzly before being conveyed from the mine to surface stockpiles. A diverter is available at the surface to allow waste rock to be stockpiled for removal and to allow stockpiling of ores for later feeding to the ROM stockpile via an emergency bin as required. An overbelt magnet removes tramp steel from the ore before it is sent to the ROM stockpile.

Four variable speed apron feeders are available to recover ore from the stockpile and feed to the primary crusher. A second overbelt magnet removes tramp steel from the primary screen feed. ROM ore is fed onto the 50 mm heavy duty primary screen from which the oversize is sent to primary crushing and the undersize direct to secondary crushing.

A variable speed vibrating feeder at the base of the primary crusher feed bin feeds the primary crusher. Primary crushed ore joins secondary crushed ore and is sent to the four sizing screen feed bins. Each bin has a variable speed vibrating feeder to feed the four sizing screens. The screens are double deck with the top deck only working to protect the bottom deck from large particle damage. Oversize from both decks join the undersize from the primary screen and feed secondary crushing. Each of the three secondary crusher feed bins has a vibrating feeder, each feeding a secondary crusher.

Sizing screen undersize is sent to the mill feed stockpile. The undersize is nominally 8 mm P_{80} to minimise the potential for scatting (discharging unground oversize) and maximise grinding efficiency in the primary mill. The mill feed stockpile is covered to minimise dust and has four vibrating feeders below it that feed ore onto the two parallel mill feed conveyors.

Milling is conducted in two identical parallel circuits, each consisting of two identical ball mills in series. A parallel configuration is preferred to manage the mine ramp up period. The primary ball mill is designed to conduct the coarse grind component only and will reduce the ore to about 150 µm P_{80}. The transfer size between the mills is selected so that all the mills are the same size and draw equivalent power. Final grinding to 53 µm P_{80} occurs in the secondary mill. To maximise grinding efficiency the second mill will use smaller grinding media than the first.

Dry mill feed from the conveyor falls into the primary mill feed chute where water is added. The new feed is joined by the primary cyclone underflow. The primary mill discharges through a trommel designed to remove ball scats (spent or broken mill balls) and directs them to a bunker for periodic removal. Mill discharge slurry passes through the trommel to the mill discharge sump. Water is added to control cyclone feed percent solids. Variable speed duty and standby pumps are available to feed the primary cyclone cluster. All the cyclone underflow returns to the primary ball mill and the overflow is sent to a linear screen. The linear screen removes any tramp oversize from the cyclone overflow to ensure secondary milling efficiency is maximised. The linear screen oversize is scalped on a static screen to remove wood, wire or other material and the undersize slurry gravitates to the secondary mill discharge sump.

New feed to the secondary milling circuit, together with secondary mill discharge, is fed to the secondary cyclone cluster by duty and standby variable speed pumps. Cyclone underflow feeds the secondary ball mill and the primary flotation collector additions are added to this stream. The secondary ball mill discharges through a trommel screen to remove ball scats and the trommel undersize gravitates to the mill discharge sump. The cyclone overflow feeds the flotation feed conditioning tank.

Roughing and scavenger flotation takes place in two parallel trains. Each train comprises of a bank of seven cells. The first two will perform the roughing duty, with the remainder scavenging. Rougher concentrate from both circuits is sent to a common cleaner circuit whilst scavenger concentrate forms part of regrind mill feed. Scavenger tails forms the majority of the final tails stream. There is a single regrind and cleaning circuit taking the primary concentrates from both trains.

The flotation feed is pumped from the conditioning tank via pumps (two operating, one standby – all Variable Speed Drive, VSD) to the rougher flotation banks. The flotation feed stream is sampled for accounting purposes. Frother and more collector are added at the feed box to the first rougher flotation cell and each can be added as required to subsequent rougher and scavenger cells. Rougher concentrate from the first two cells is pumped (duty and standby) to rougher cleaning cells. Rougher tails feed the first scavenger cell.

Rougher cleaner concentrate is sent to rougher recleaner flotation and the coarse component of the final concentrate is produced. The rougher recleaner concentrate is pumped to the concentrate thickener. Tails from both the rougher cleaner and rougher recleaners are sent to regrind milling.

The three regrind mill feed streams (scavenger concentrate, rougher cleaner tails and rougher recleaner tails) are pumped to the regrind feed tank. Regrind circuit feed is pumped (duty and standby) to the regrind densifying cyclones. Densifying cyclone overflow reports directly to the regrind product tank and cyclone underflow is fed to the regrind mills. Reground material reports to the Regrind Product Tank. Regrinding is planned to be conducted to 10 µm P_{80}. The reground product is sampled and its particle size is continuously measured for control purposes.

Reground material is pumped to the scavenger cleaner flotation conditioning tank (duty and standby pumps). Reagents are added and the slurry is then pumped (duty and standby) to the scavenger cleaner flotation bank. Scavenger cleaner concentrate is pumped to scavenger recleaning and scavenger cleaner tails form part of final tails. The scavenger cleaner concentrate is pumped to scavenger recleaner flotation.

Scavenger recleaner concentrate is pumped to the concentrate thickener feed tank and scavenger recleaner tails are pumped to the final tailings thickener.

The two final concentrate streams are mixed in the concentrate thickener feed tank. Flocculant is added as concentrate-flows to the thickener feed well by gravity. Thickener overflow reports to the concentrate thickener overflow tank, from where it is distributed to the process water circuits. Thickener underflow is pumped (duty and standby) to the filter feed tank and it is sampled for accounting purposes.

All three tailings streams (scavenger tails, scavenger cleaner tails and scavenger recleaner tails) report to the tailings thickener feed tank. Flocculant is added as the slurry flows by gravity to the tailings thickener feed well. All tailings thickener overflow reports by gravity to the process water tank. Tailings thickener underflow is pumped (duty and standby) to the tailings pumping tank and it is sampled for accounting purposes. Multistage slurry pumps send the slurry to the tailings storage facility.

Concentrate is filtered and then sampled and bagged for transport to customers.

The design criteria are shown in Table 17.1. The two availability figures are in line with industry norms for these types of operations after incorporating allowances for local issues such as power reliability.

Table 17.1 Design Criteria

	Unit	Value (Design)
Annual Plant Feed	tpa	6,000,000
Overall Crusher Availability	%	75
Crusher Operating Time	hpa	6,570
Crushing Circuit Feed Rate	t/h	913
Overall Mill Availability	%	87
Mill Operating Time	hpa	7,598
Milling Circuit Feed Rate	t/h	790

17.3 Concentrator Basis of Design

The concentrator design is based on expectations for the first nine years of operation (Table 17.2). The ROM feed is taken to be 89% Hypogene and 11% Supergene based on the total production schedule average. Appropriate design margins have been incorporated.

Table 17.2 Concentrator Basis of Design

Option	Units	Value	Comment
Flotation Feed	Mtpa	6	–
Average Feed Rate	t/h	790	–
Maximum Feed Rate	t/h	869	+10%
Average Feed Grade	% Cu	3.81	Kamoa PFS Mine Plan
Early Ore Grade	% Cu	4.26	Average (Years 1–10)
Design Feed Grade	% Cu	5.14	Max Annual Average Grade Years 1–10 plus 13%
Relative Abundance - Hypogene (%)	Mass %	89	Kamoa PFS Mine Plan
Relative Abundance - Supergene (%)	Mass %	11	Kamoa PFS Mine Plan
Concentrate Grade	% Cu	37	Lower than mine production assumption, see text
Copper Recovery	%	86	From testwork
Design Mass Pull	Mass %	11.9	Based on Design Feed Grade

The ability to blend feed from multiple sources underground should provide the project with a high degree of control over plant feed grade and as such the maximum head grade expected has been chosen to be only marginally higher than the highest annual average grade.

The concentrate grade of 37% Cu was chosen based on a high chalcopyrite feedstock and represents a worst expected case in terms of tonnes to be thickened, filtered and bagged. A grade of 39% Cu has been used in mine planning and this is a legitimate average design grade based on testwork results.

17.4 Flow Diagrams

The block flow diagram for the crushing and milling circuit is shown in Figure 17.1.

Figure 17.1 Kansoko Crushing and Milling

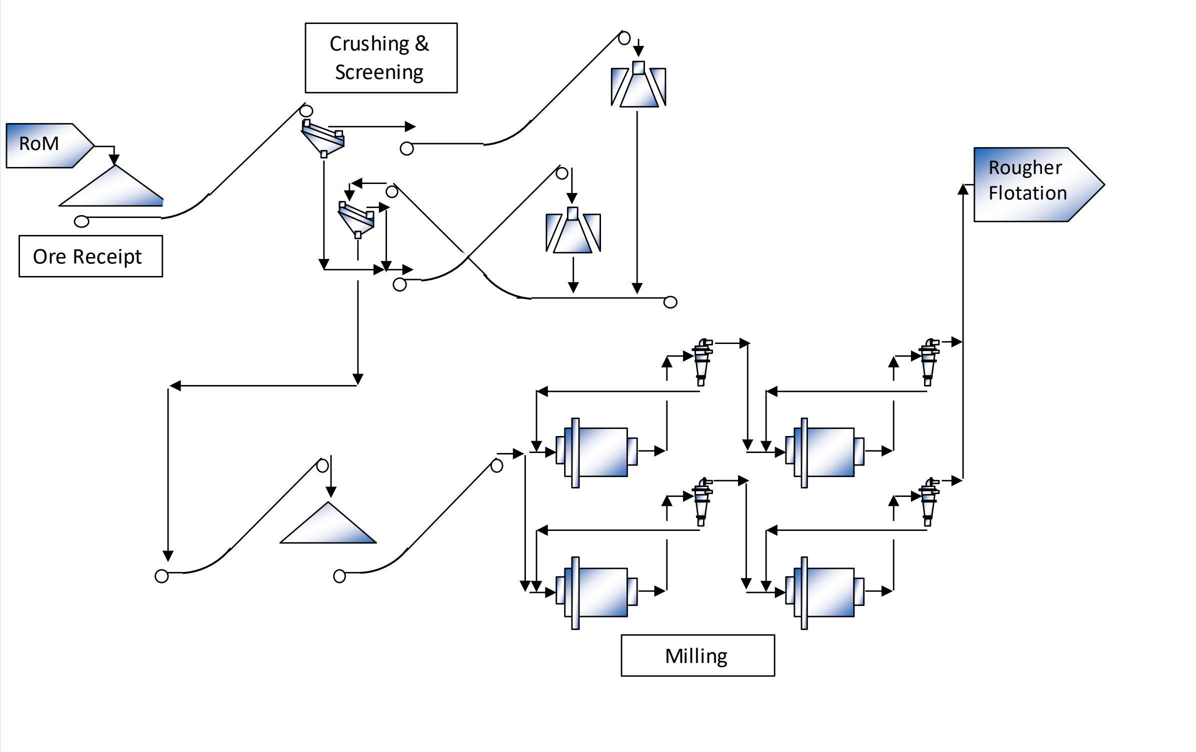


Figure courtesy MDM, 2017.

The block flow diagram for flotation, concentrate handling and tailings is shown in Figure 17.2.

Figure 17.2 Kansoko Flotation Circuit

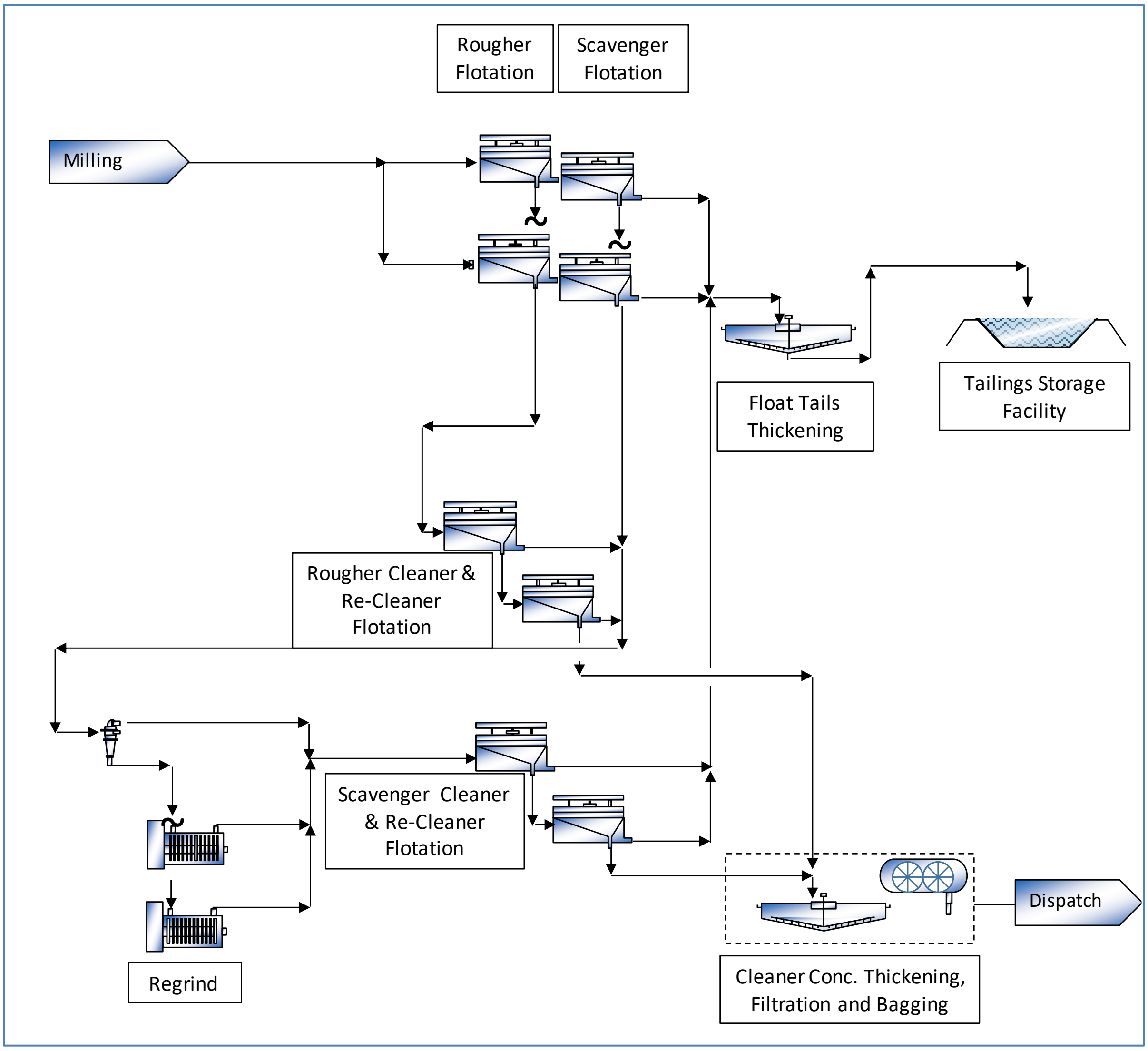


Figure courtesy MDM, 2017.

17.4.1 Reagents, Services and Utilities

Reagent plants, located close to the flotation circuit, provide for the mixing and supply of the necessary reagents for flotation and flocculants for thickening.

All floatation cells are forced air and dedicated blowers supply manifold air for the flotation cells.

Raw water from a wellfield is pumped to a raw water dam. Filtration and treatment plants use the raw water to produce a range of water qualities as required for potable water, gland seal water, fire water and process water usage. Distribution systems for each water type are included, ensuring delivery of sufficient quantity at the required pressure.

Compressed air is supplied and distributed for the use of general plant requirements and filters. A dried air (dew point <0°C) supply is available for air actuated instruments and valves.

17.4.2 Concentrator Equipment Specifications and List

Table 17.3 provides a summary of the major mechanical equipment for the proposed concentrator. This list forms the basis of a much more detailed concentrator capital cost estimate.

Table 17.3 Concentrator Equipment Requirements Summary

Item	Description	Size/Capacity	No. Required + standby	Power Installed kW per unit
Crushers	Primary cone	CS660	2	315
	Secondary cone	CH865	3	500
Screens	Primary	2.4 m x 4.27 m	2	45
	Secondary	3.1 m x 6.1 m	4	55
Mills	Primary Ball Mill	22 ft x 36 ft	2	7,000
	Secondary Ball Mill	22 ft x 36 ft	2	7,000
	Concentrate regrind	IsaMill M10000	2	3,000
Cyclones	Primary cluster	750 mm Diameter	4 + 1	500 (1+1 feed pump)
	Secondary cluster	420 mm Diameter	9 + 1	355 (1+1 feed pump)
	Concentrate regrind cluster	165 mm Diameter	15 + 1	75 (1+1 feed pump)
Blowers	Flotation air	65 700 Nm^3/h @ 150 kPa	4 + 1	200
Flotation cells (includes agitators)	Rougher	320 m^3	4	280
	Scavenger	320 m^3	10	280
	Rougher cleaner	50 m^3	5	75
	Rougher recleaner	30 m^3	6	45
	Scavenger cleaner	160 m^3	6	160
	Scavenger recleaner	30 m^3	6	45
Thickeners	Concentrate	20 m Diameter	1	11
	Tailings	50 m Diameter	1	18
Filters	Concentrate	Hyperbaric Disc 57216-2L	2	55
Tailings Pumps	Centrifugal	840 m^3/h	4 + 4	185

Table 17.4 lists the estimated projected water, consumables and power requirements for the concentrator.

Table 17.4 Projected Concentrator Water, Power, and Consumables

Item	Description	Consumption per tonne of Plant Feed	Annual Requirement
Power	Electric	58.8 kWh/t	353 GWh
Water	Raw make-up	0.5 m^3/t	3,025 ML
Reagents	Frother	95 g/t	570 t
	Collector	156 g/t	936 t
	Promotor	28 g/t	168 t
	Flocculant (Tailings and Concentrate)	35 g/t	210 t
Consumables	Grinding media (75 mm steel balls)	0.88 kg/t	5,296 t
	Grinding media (35 mm steel balls)	1.34 kg/t	8,034 t
	Grinding media (2 mm Ceramic)	68 g/t	408 t

Most consumables are supplied in bulk bags or containers. A kibble is used to load grinding media into the ball mills. The low abrasion index of the ore (Ai = 0.08) ensures that ball consumption will be relatively low compared to most similar projects.

17.5 Processing Production Schedule

The processing production schedule is shown in Table 17.5. The schedule is driven more by availability of ore than by plant capacity. The underground mines are being developed in Years –2 to 4, reaching the full 6 Mtpa capacity in Year Four. In Years 1 to 3 the plant will typically require only one line to operate. Alternatively, there may be options for the full plant to run on a reduced daily time schedule.

Table 17.5 Processing Production Schedule

Description	Unit	Total	Project Time (Years)													
			–1	1	2	3	4	5	6	7	8	9	10	11	12	13
Ore Milled	(kt)	125,182	-	1,169	2,608	3,205	4,667	6,014	6,028	6,010	6,015	6,027	6,027	6,027	6,050	6,007
Cu Grade Milled	(% Cu)	3.81	-	3.45	3.99	4.40	4.42	4.26	4.21	4.55	4.39	3.97	3.75	3.85	3.99	3.71
Concentrate Produced	(kt)	11,405	-	97	243	338	498	613	614	679	648	587	558	564	583	539
Concentrate Cu Grade	(% Cu)	37	-	36	37	37	37	37	37	36	36	36	36	36	36	36
Cu in Concentrate	(kt)	4,178	-	35	91	124	183	226	224	245	236	212	201	205	213	196
Cu in Concentrate	(Mlb)	9,211	-	78	200	274	403	498	495	541	520	468	443	452	469	432
Description	**Unit**	**Total**	**Project Time (Years)**													
			14	**15**	**16**	**17**	**18**	**19**	**20**	**21**	**22**	**23**	**24**	**25**	**26**	
Ore Milled	(kt)	125,182	6,077	6,022	6,005	6,019	6,012	6,009	6,018	6,022	4,417	3,126	1,858	943	802	
Cu Grade Milled	(% Cu)	3.81	3.85	3.76	3.46	3.17	3.12	3.24	3.44	3.86	3.49	3.42	3.89	3.83	2.93	
Concentrate Produced	(kt)	11,405	560	544	485	436	431	448	490	551	355	253	168	76	49	
Concentrate Cu Grade	(% Cu)	37	36	36	37	37	37	37	37	37	37	37	37	38	38	
Cu in Concentrate	(kt)	4,178	204	198	178	162	160	168	180	203	132	93	63	29	19	
Cu in Concentrate	(Mlb)	9,211	449	435	392	356	353	369	398	447	290	205	138	64	41	

17.6 Comments on Section 17

This plant design is based on the flowsheet used in the Kamoa PFS in 2015. The laboratory flotation flowsheet used in the plant design has been applied a number of times since the PFS was completed at two different laboratories and has proven to give acceptable results for a variety of ore types. This recent work has provided confidence that the liberation characteristics of the ores in the Kamoa and Kansoko areas of the copper system are similar and respond well in identical test conditions. In addition, comminution testing shows that ores from all areas have similar breakage characteristics and will respond in a similar fashion during crushing and grinding. Overall, no flowsheet risks arose as a result of testing the various different feeds.

ROM ore is assumed to have a topsize of 300 mm, controlled by intensive blasting and 300 mm square grizzly installations at each ore dump point underground. If this topsize control is found to be unmanageable by blasting alone, then additional underground crushing may be required. Note that underground grizzly sizes can only be relaxed with caution as particles in excess of 300 mm will cause problems for the conveying systems that bring material to the surface from underground.

The plant design is based on a 53 µm flotation feed P_{80} and a 10 µm regrind P_{80} of the flotation middlings. Testing has shown these parameters to be robust and good control has been gained in the more recent testwork over silica content of the final concentrate.

The flotation circuit configuration deliberately avoids recycle streams in accordance with the XPS testing philosophy. This results in (at least theoretically) well-defined residence times throughout the circuit. However, it presents a risk with regard to managing varying ore grade and copper sulphide mineralogy. The most likely stream to be recycled in the current configuration is the scavenger recleaner tail (recycle to scavenger cleaner feed). Another possibility is to recycle rougher recleaner tail to rougher cleaner feed as this may reduce the regrind duty. Flowsheet provision for the scavenger recycle is allowed and consideration should be given to making provision for the rougher recycle.

The copper mineralisation determines how much copper is recoverable by flotation and the grade of concentrate that can be generated. The mineralisation is highly variable and further work is needed to better define mineralogy in the various parts of the Kamoa deposit. Good quality mineralogical information will be necessary for feed grade blending and controlling final concentrate grade. Even higher quality information is required if an on-site smelter is constructed in the future of the project.

In the dominant hypogene ores concentrate grade will vary with the relative chalcopyrite, bornite, chalcocite and digenite proportions and also with the amount of silica recovered to concentrate. One variable, the silica content of the concentrate, is now shown to be controlled in the test procedure. The variable copper mineralogy will also need to be understood and may be able to be inferred from Cu:S ratio. Samples with varying Cu:S and fully defined copper mineralogy must be subjected to variability testing, which is in plan.

Hypogene ore copper recovery is always high as there is little unfloatable copper mineralisation present. Improved hypogene ore definition during planning and production requires a means of identification of the relative proportions of the important copper minerals in core and drill cuttings and this should also be explored in the next study phase.

Supergene ore has broad variability in both the relative proportions of the floating copper minerals (as described for hypogene in the previous paragraph) and the proportion of copper that will be lost to tailings in non-floating minerals. This leads to uncertainty with regard to both copper recovery and grade of copper concentrate. Improved supergene ore definition requires identification of the relative proportions of chalcopyrite, bornite, chalcocite, diginite and covellite (floating copper minerals) together with a measure of how much copper is in non-floating species such as native copper, azurite, malachite and cuprite. The overall project impact of the supergene uncertainty is important, but it is not felt to be material to the project economics. Supergene only represents 11% of the concentrator feed in the Kansoko mine plan and supergene will always be blended with hypogene in the mill feed.

18 PROJECT INFRASTUCTURE

18.1 Introduction

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

This section describes the project infrastructure work that was developed for the Kamoa 2017 PFS. The project infrastructure includes power supply, tailings dams, communications, logistics, transport options, materials, water and waste water, buildings, accommodations, security, and medical services.

It is currently anticipated that in the first five years of production concentrate will be transported via road to Ndola in Zambia and thereafter via rail to ether Richards Bay or Durban harbour in South Africa. After five years it is assumed the Angolan rail line will be operational and a rail spur will be constructed to Kamoa for direct rail transport to the Angolan port of Lobito.

Power for the Kamoa-Kakula Project is planned to be sourced from three hydro power plants: the Koni, Mwadingusha and Nzilo power stations. All three stations require refurbishing. The three plants combined could produce over 200 MW. Prior to completion of the refurbishments, development and construction activities at the mine will be powered by electricity sourced from the grid and on-site diesel generators.

18.2 Site Plan and Layout

A plan showing the locations of the mines and key infrastructure for Kakula and Kansoko mines is shown in Figure 18.1. Figure 18.2 displays the locations of the proposed Kamoa plant site, and closely associated facilities.

Figure 18.1 Kamoa-Kakula 2017 Development Plan Site Plan

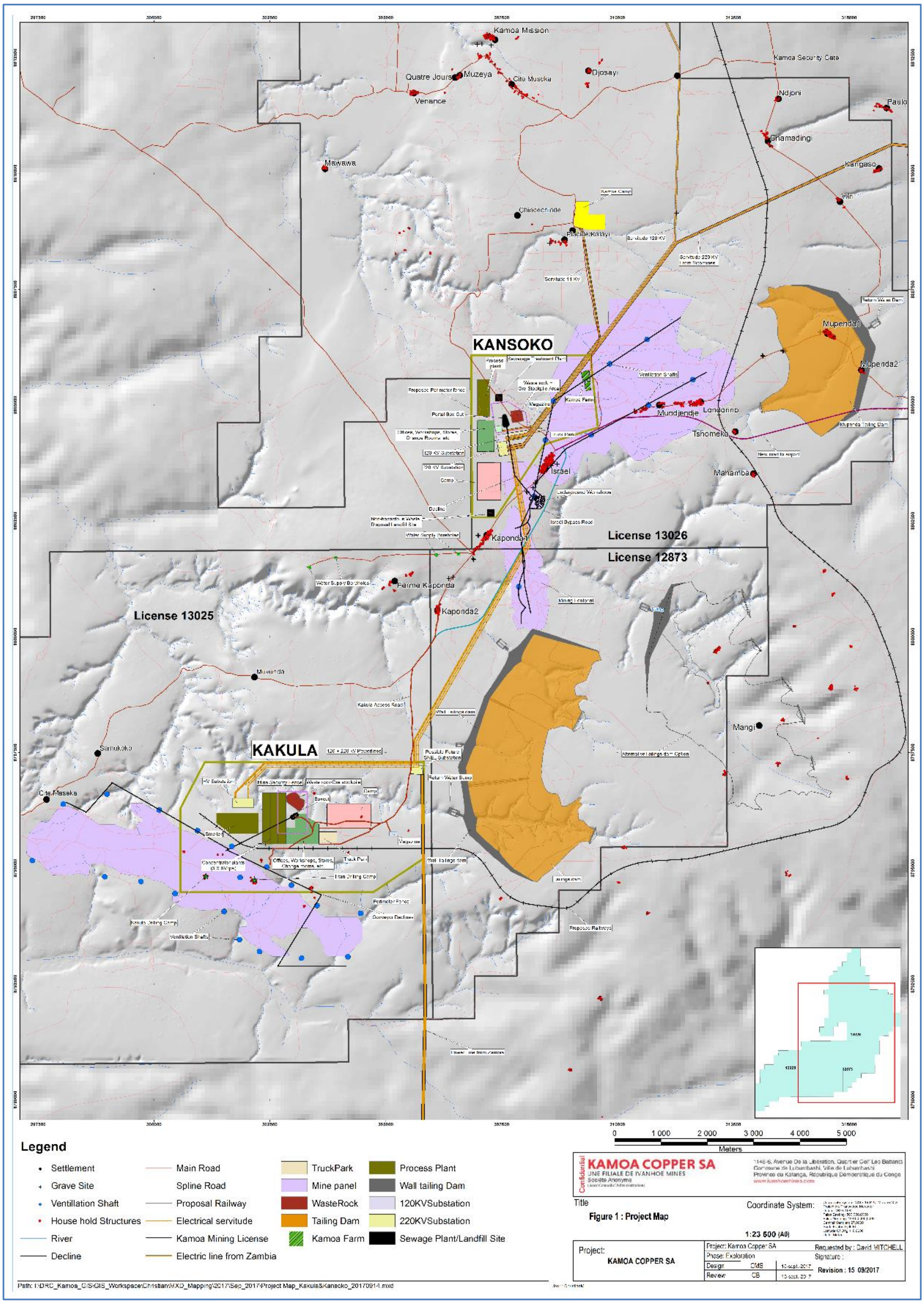


Figure by Kamoa Copper SA, 2017.

Figure 18.2 Site Conceptual Infrastructure Layout Plan

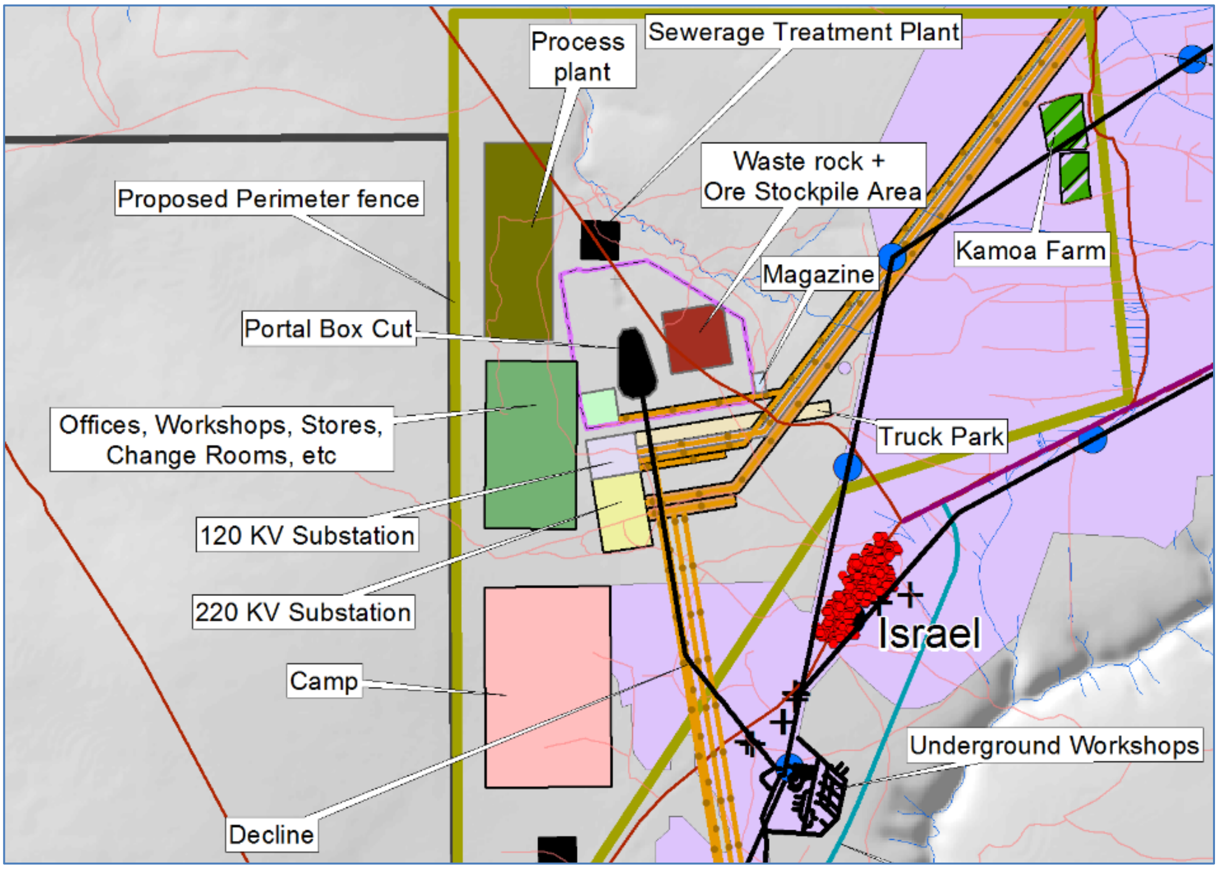


Figure by Ivanhoe, 2017.

The site is compact and incorporates the process plant, utilities, reagent preparation, laboratory, offices, construction camp, electrical infrastructure, water infrastructure, surface mining offices and workshops, vehicle parking, warehouse storage, concentrate storage and lay-down facilities.

All infrastructure has been incorporated in the capital cost estimate.

18.3 Power

18.3.1 Generation

Power for the Kamoa-Kakula Project is planned to be sourced from the DRC's state-owned power company Société Nationale d'Electricité (SNEL), electrical interconnected grid. This electrical grid faces a shortage of power generation due to ageing hydropower plants with a number of non-working turbines that require repair.

The hydro power plants in the SNEL southern grid that are considered in the Ivanhoe SNEL power project are: Koni, Mwadingusha and Nzilo. All three require refurbishing. The three plants combined could produce over 200 MW. Prior to completion of the refurbishment, development and construction activities at Kamoa will be powered by electricity sourced from the SNEL grid and on-site diesel generators.

In June 2011, Ivanhoe signed a Memorandum of Understanding (2011 MOU) with SNEL. The 2011 MOU led to the signing of a pre-financing agreement with SNEL in June 2012 under which Ivanhoe pledged a loan of USD 4.5 million for the emergency repair of generator unit 1 at Mwadingusha hydroelectric power station. This will unlock 10 MW of power required for development and construction activities.

After subsequent negotiations, SNEL granted Ivanhoe an exclusive right to conduct full rehabilitation on the Mwadingusha and Koni plants following completion of a feasibility report on the work. A study to rehabilitate the Mwadingusha and Koni power plants was carried out by Stucky Ltd in 2013 (Stucky Report on Mwadingusha and Koni).
On 14 March 2014, SNEL and Ivanhoe signed a Financing Agreement for the rehabilitation of the two power stations and associated high voltage infrastructures. This financing agreement is in the form of a loan to SNEL that will be re-paid with 40% discounted power tariffs.

After rehabilitation of these two hydro power plants for the first phase mine development, SNEL guarantees 100MW to Kamoa. In addition to the generating plant refurbishment, a new 220 kV substation for the new 220 kV high-voltage line to the mine site is also required to provide power to the project. This supply will feed into the new 220/11 kV substation from where the process plant will receive its power.

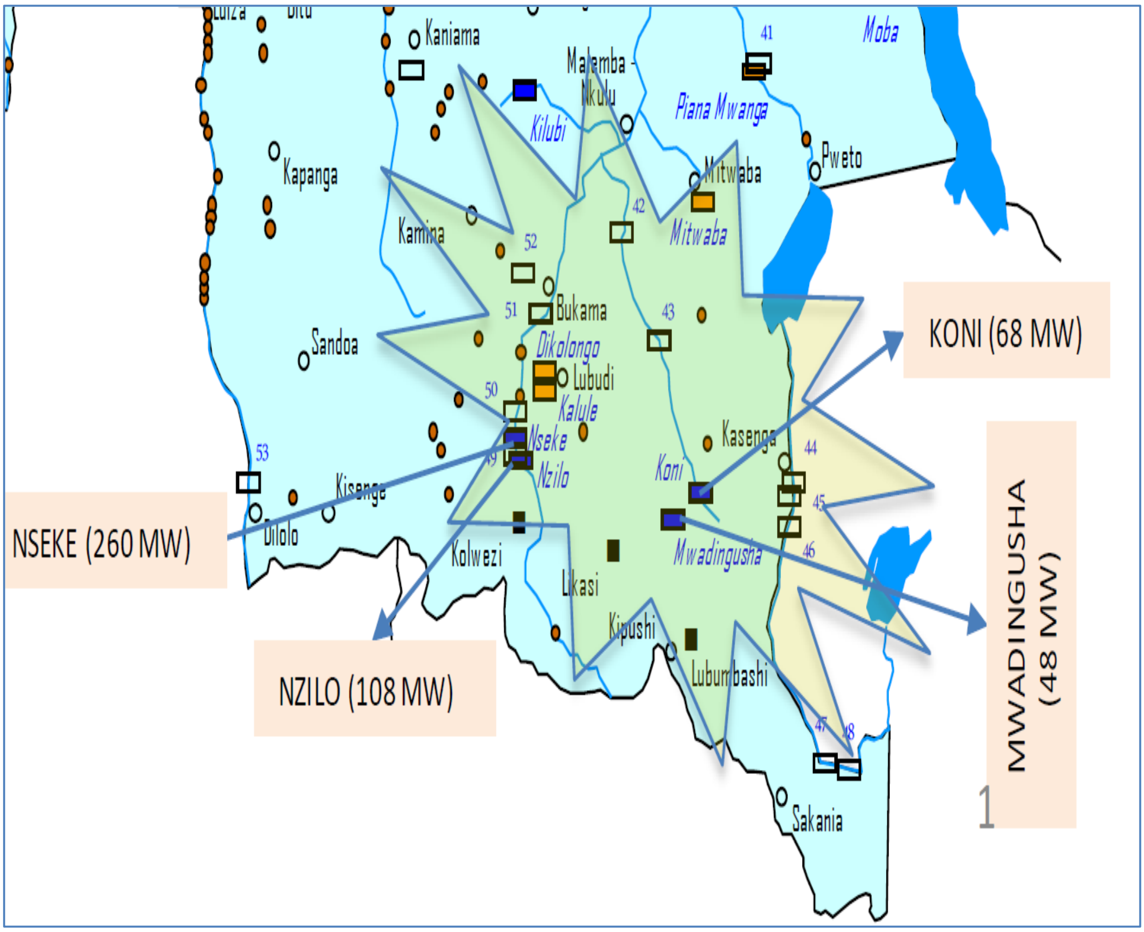
For construction power (10 MW), a 120 kV high-voltage spur line (20 km long) has been built to tap power from the RO-Kisenge line to the Kansoko Mine. RO is the acronym for "Répartiteur Ouest" i.e. Western Dispatch substation in Kolwezi.

A 120/11 kV, 15 MVA mobile substation (Figure 18.3) has also been installed and commissioned to feed construction power. The line and substation will be retained as emergency back-up power supply after the commissioning of the main 220 kV supply line and substation. Figure 18.4 shows the HV Mobile Substation installation.

Diesel generators for back-up power have been installed and are operational. The diesel generator capacity will be increased in size to ultimately provide the mine and plant with the required standby power.

In 2013, Ivanhoe signed an additional Memorandum of Understanding (2013 MOU) with SNEL to upgrade a third hydroelectric power plant, Nzilo 1. A study to rehabilitate the Nzilo 1 power plant was carried out by Stucky Ltd in 2014 (Stucky Report on Nzilo 1). It is proposed to upgrade the Nzilo 1 hydroelectric power plant to its design capacity of 100 MW.

Figure 18.3 Power Plants Locations



Map by Ivanhoe, 2017.

Figure 18.4 HV Mobile Substation Installed at Kansoko, October 2016

Photograph by MDM, 2016.

Mwadingusha Hydroelectric Power Plant

The Mwadingusha hydro power plant is located on the Lufira River, approximately 70 km from the city of Likasi in the province of Haut-Katanga in the DRC. The hydro facility was built in 1928 and comprises six turbines with an installed generation capacity of 71 MW at a gross hydrostatic head of 114 m. Turbines four and five were installed in 1938, whilst turbine six was installed in 1953. Of the turbines installed, turbines four, five, and six, are currently operational.

Koni Hydroelectric Power Plant

Koni is located 7 km downstream of Mwadingusha and was built in 1946 with an installed generation capacity of 42 MW at a hydrostatic head of 56 m. The turbine hall comprises three turbines, only turbines one and two are currently operational.

Nzilo Hydroelectric Power Plant

The Nzilo hydro power plant is located on the Lualaba River, approximately 30 km from the city of Kolwezi in the DRC province of Lualaba. The hydro facility was built in 1952 and comprises four turbines with an installed generation capacity of 108 MW at a gross hydrostatic head of 74.5 m. Three out of four turbines installed are currently operational but need to be renewed.

18.3.2 Transmission and Substations

The power plants substations and lines will be refurbished. A new Gas Insulated Substation (GIS) 120/6.6 kV substation will be built at Mwadingusha hydro power plant. Koni and Nzilo hydro power plants substations will be refurbished completely. OPGW (optical ground wire) will be installed to the two Nzilo-RO 120 kV lines (20 km).

The two 120 kV bays at RC substation in Likasi where Koni and Mwadingusha power connect to the SNEL grid will be also refurbished.

In the interim or first phase, 10 MW can be supplied to the Project over a new 20 km transmission line from the RO-Kisenge line to Kansoko Mine for construction power, through a 120/11 kV, 15 MVA mobile substation that is installed at Kansoko Mine.

At the second phase, in order to achieve high power availability over the longer-term, a new double 220 kV circuit transmission line (20 km) will be constructed to feed power to the 220/11 kV Kansoko Mine substation. A new SNEL 220 kV sub-station (named NRO, Nouveau Répartiteur Ouest) will be constructed adjacent to the existing 220 kV substation owned by Sicomines mine west of Kolwezi. NRO will be fed from the SCK substation in Kolwezi, which is a major transmission hub in SNEL's southern network, connecting to the Northern network and the Inga power plant via a 1,200 km DC line.

The Figure 18.5 shows the design of new transmission lines and substations.

Figure 18.5 Planned Transmission Lines and Substations

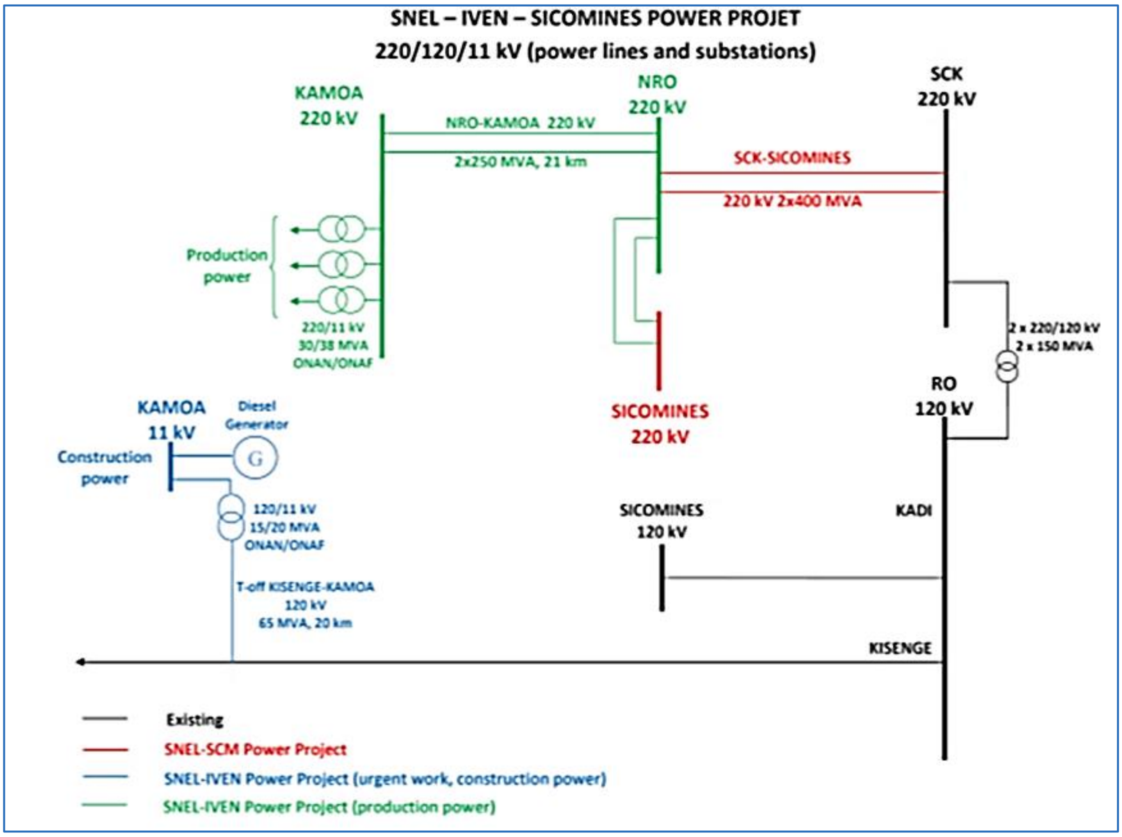


Figure by Ivanhoe, 2016.

18.4 Tailings Storage Facility

Epoch Resources (Pty) Ltd (Epoch) completed a basic design of the Tailings Storage Facility (TSF) and associated infrastructure as part of the Kamoa 2017 PFS.

The terms of reference that Epoch was responsible for include:

- A Tailings Storage Facility (TSF) that accommodates 109,480,000 dry tonnes of tailings over a 26-year LOM.
- A Return Water Dam (RWD) and Return Water Sump (RWS) associated with the TSF.
- The associated infrastructure for the TSF (i.e. perimeter slurry deposition pipeline, stormwater diversion trenches, perimeter access road etc.).
- Revalidation of the Mupenda site for the higher production rate (i.e. 6 Mtpa mill feed).
- Estimation of the capital costs to an accuracy of ±25 percent, operating costs associated with these facilities to an accuracy of ±25 percent and closure costs to an accuracy of ±35 percent.
- Estimation of the costs over the life of the facility.

The site selection study undertaken, by Epoch, found the most favourable site as being the Mupenda site.

Figure 18.6 Mupenda TSF in Relation to Kansoko

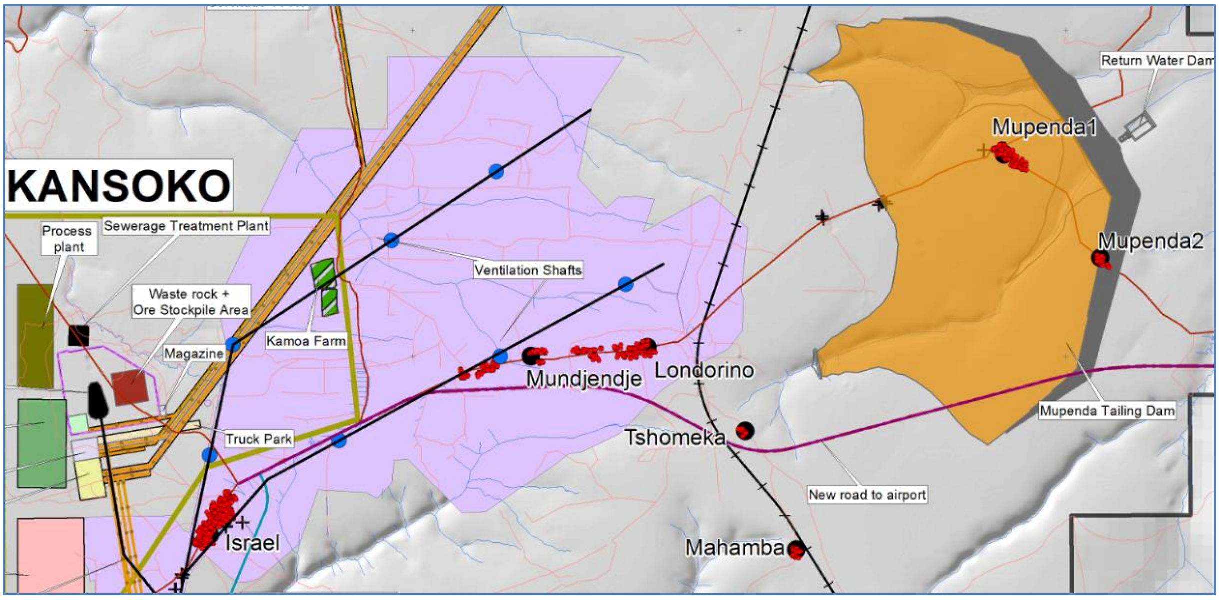


Figure by Ivanhoe, 2017.

The key design features of the TSF are as follows:

- The TSF will be constructed as a double valley impoundment dam with a compacted earth impoundment wall. This will have the following features:
  - The TSF impoundment walls will be constructed as a downstream facility.
  - The wall is to be raised in 7 phases, where Phase 1 is at elevation 1465 m amsl and the last phase is at elevation 1495 m amsl.
  - A final phase (Phase 8), comprising a smaller upstream impoundment wall will be constructed on top of Phase 7 when the rate of raise reduces to <1.0 m/year.
  - The TSF has a total footprint area of 540 Ha, a maximum height of 52 m and a final rate of rise of <1.0 m/year.
- A Return Water Dam with a storage capacity of approximately 45,000 m^3.
- A concrete lined Return Water Sump with a water storage capacity of 2,000 m^3.
- A slurry spigot pipeline along the crest of the TSF.

18.4.1 Project Location

The terrain is mostly grasslands with some dense pockets of trees. The general topography of the Mupenda site area can be seen in Figure 18.7.

Figure 18.7 General Topography of the Preferred Kamoa TSF Area

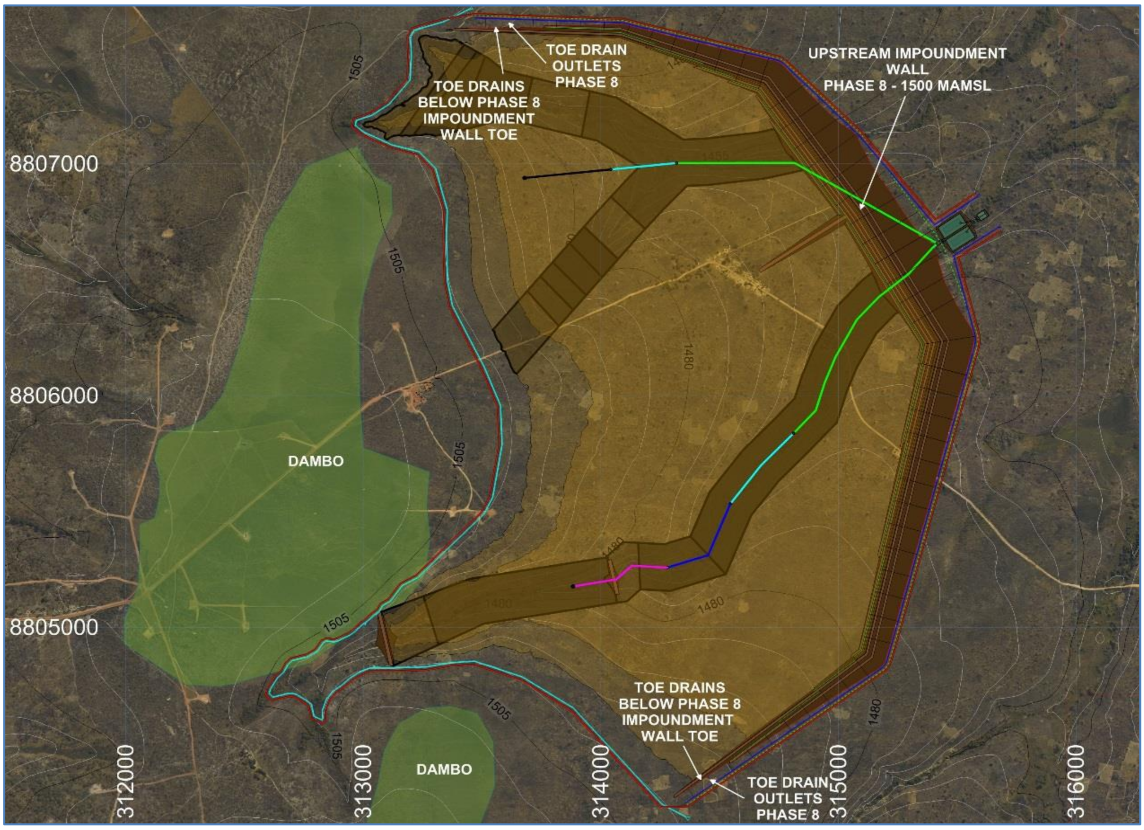


Figure by Epoch, 2017.

18.4.2 Design Criteria and Assumptions/Constraints

The design of the TSF was based on the design criteria shown in Table 18.1.

Table 18.1 Design Criteria Associated with Kansoko TSF

Description	Value	Unit
Design Life of Facility	22	years
Tailings Deposition Rate:		
Year 1	1,380,000	Dry tpa
Year 2	1,840,000	Dry tpa
Year 3	2,760,000	Dry tpa
Year 4	4,140,000	Dry tpa
Year 5–Year 22	5,520,000	Dry tpa
Particle SG of Tailings product	2.85	
In-Situ Void Ratio	1.0	
Particle size distribution of Tailings product	80% passing 53 micron	
Placement Dry Density of Tailings:		
Sub-aqueous	1.3	t/m^3
Sub-aerial	1.5	t/m^3
Average	1.4	t/m^3
Site's Seismicity	0.08 g	PGA

The assumptions adopted for the TSF are as follows:

- Sufficient and suitable construction materials for the preparatory earthworks associated with the TSF can be sourced from the TSF basin and nearby borrow pits.
- The legislation that has been adopted for the purpose of this study is "Appropriate Best Practice Measures" which have been taken to be the South African Tailings Disposal Facility Design Standards and Codes (i.e. SANS 0286:1998 – "Code of Practice for Mine Tailings"). DRC laws regulating TDFs have been considered in the design as discussed in Section 18.4.4.
- The tailings have been classified as a "leachable mine waste" by Golder Associates (Pty) Ltd (Golder). Therefore, according to the DRC Regulations, in areas where the basin of the TSF has a permeability greater than 1 x 10-6 cm/s, the TSF must be lined with an appropriate liner system.

18.4.3 Climatic Data

For the purpose of this study it was assumed that the rainfall and the evaporation for the Kamoa site and Kolwezi (20 km east of Kamoa) are similar. This assumption was made because no long-term rainfall information (Table 18.2) was available for the Kamoa site. The Kolwezi rainfall and evaporation data were sourced from the Kolwezi weather station.

Table 18.2 Average Monthly Rainfall and Lake Evaporation Values for Kolwezi

Month	Average Rainfall (mm)	Lake Evaporation (mm)
January	211	73.2
February	189	92.4
March	216	129.4
April	80	143.2
May	10	177.1
June	2	176.3
July	0	164.8
August	3	112.4
September	23	109.3
October	90	146.3
November	180	101.6
December	209	89.3
Annual	1213	1515.3

Table 18.3 Design Storm Rainfall Depths for Kolwezi

Duration (Days)	Rainfall Depth (mm) For Each Recurrence Interval						
	2 Years	5 Years	10 Years	20 Years	50 Years	100 Years	200 Years
1	48	55	59	62	66	69	71
2	88	100	109	115	122	127	132
3	104	118	128	135	144	150	155
4	112	127	138	146	155	161	167
5	115	130	141	149	158	165	171
6	117	130	145	153	162	169	175
7	120	137	148	156	166	172	178

18.4.4 DRC Regulations Pertaining to Tailings Storage Facilities

The local DRC laws regulating TSF stipulate the following:

- Site investigation techniques: for determining the properties of materials such as backfill, foundations and other structures, as well as methods to put such structures in place and compacting methods must be carried out in accordance with state of the art methods.
- Appropriate stability calculations must be undertaken and shall take into consideration long-term conditions that might affect structures, including static and dynamic loads.
- An appropriate seismic coefficient must be used for the seismic stability analyses i.e. a seismic coefficient with an annual probability of exceedance of 1 in 476 years (10% over a period of 50 years) for sites with non-acid generating material, and an annual probability of exceedance of 1 in 1,000 years for sites with acid generating material.
- The slope stability's factor of safety should be greater than 1.5 for static stability analyses, and between 1.1 and 1.3 pseudo-static analyses.
- Appropriate measures must be taken to ensure that no toxins from any tailings storage areas enter into the groundwater. Different requirements are applicable depending on the geochemical nature and toxicity of the tailings product.
- Surface erosion problems shall be controlled by preferably planting vegetation. Erosion problems in unconsolidated materials shall be eliminated by reducing the hydraulic gradient. If materials of different particle gradings are placed in contact with each other, appropriate filter criteria must be observed.

18.4.5 Liner Requirements

Golder's geotechnical division was tasked with assisting with the preliminary specification of the liner solution at the Mupenda TSF site. This was documented in their report titled: "Geotechnical and Geochemical Aspects of the Liner Recommendation for the Mupenda TSF". They identified a layer of Kalahari sand along the valley which does not provide the necessary permeability requirement to adhere to the DRC Regulations. This sandy layer was found to have a permeability higher than the limit of 1x10-6 cm/s. The remainder of the footprint was found to adhere to this permeability limit. Therefore, only the area underlain by Kalahari Sands will require a liner.

Golder investigated a Bentonite-enriched Compacted Earth Liner (BCEL) as a solution to this. However, this will require further testing to determine whether it will perform as anticipated. Epoch has compared the costs associated with a 1500 µm HDPE liner solution and the BCEL and determined that there is little difference in the costs. Therefore, a HDPE liner has been adopted for the design of the TSF and will be reviewed in the next phase of the project with the subsequent testing of the BCEL.

18.4.6 TSF Site and Design

The preferred TSF site is the Mupenda site. The site selection study was performed in 2014 by Epoch and documented in their report titled: "Site Selection Report – Kamoa Copper Project, TSF for a 3 Mtpa Plant – Addendum". This site was chosen for the TSF for the following reasons:

- The topography and soil properties are such that it will not require expensive measures to both contain the tailings and prevent ground water and surface water contamination.
- The risks associated with this site were deemed the lowest out of the other options due to the contaminated catchment downstream of the TSF due to the historical failure of the Potopoto TSF, as well as little to no people residing downstream of the site.
- Lowest construction cost of all the options considered.

18.4.7 Design Considerations

The design of a TSF usually begins with determining what type of facility will be selected to contain the tailings. Two common types of facilities are self-raised and full containment which have the following characteristics:

- A self-raised facility utilises the tailings itself to build the outer walls or "daywalls" (as this generally occurs during the day). The daywall is constructed higher than the tailings in the basin in order to maintain a minimum freeboard.
- A full containment facility utilises imported material to construct an impoundment wall and depositing tailings behind this wall.

The Kamoa tailings were investigated in 2014 and it was found that a self-raised facility would have to be constructed at a low Rate of Rise (RoR) of 1.0 m/year for safe operation. This was due to the fineness of the tailings (80% passing 53 µm). The cost comparison of self-raise vs full containment showed that at this RoR there would not be a significant cost saving in selecting self-raising.

Subsequently, the tailings Particle Size Distribution (PSD) has become even finer because 70% of the tailings is produced at a P_{80} of 53 µm (scavenger tailings) and the remaining 30% has a P_{80} of only 10 µm (scavenger cleaner and recleaner tailings). The combined tailings P_{80} is calculated to be approximately 40 µm and it will be bimodal, in accordance with the two flotation tailing sources. This means that the RoR would have to be even lower to build a self-raised TSF.

The fineness of the Kamoa tailings also has the following implications:

- Subsoil drains cannot be constructed in the tailings as they will blind and become inoperable. A full impoundment facility can utilise a curtain drain to reduce the phreatic surface through the wall. A curtain drain will not be in contact with the tailings and thus cannot blind.
- Slurry water will contain suspended solids if it is not allowed sufficient time to settle. This can be done using a silt trap/settling facility or maintaining a pool for longer durations on the TSF. A self- raised facility has the implication that it cannot hold large quantities of water as it will affect the stability, whereas a full containment does not.

The beach slope of the TSF is expected to be very flat, making pool control very difficult for an operator.

The Kamoa TSF will be constructed as a full containment facility at the Mupenda site.

18.4.8 Stage Capacity and Site Development Strategy

The impoundment wall has been phased in order to delay capital expenditure as far into the life of the facility as possible. In order to effectively phase other construction items (such as the penstock pipeline and liner), one intermediate wall which will hold the tailings back while allowing the contractor to construct and install the penstock and liner, as well as a final back wall, will be provided at the upstream side of the southern valley. The penstock, liner and intermediate and back walls will have several stages which correspond to certain impoundment wall phases (see Figure 18.8).

Figure 18.8 Impoundment Wall and Self-Raise Lift Phasing

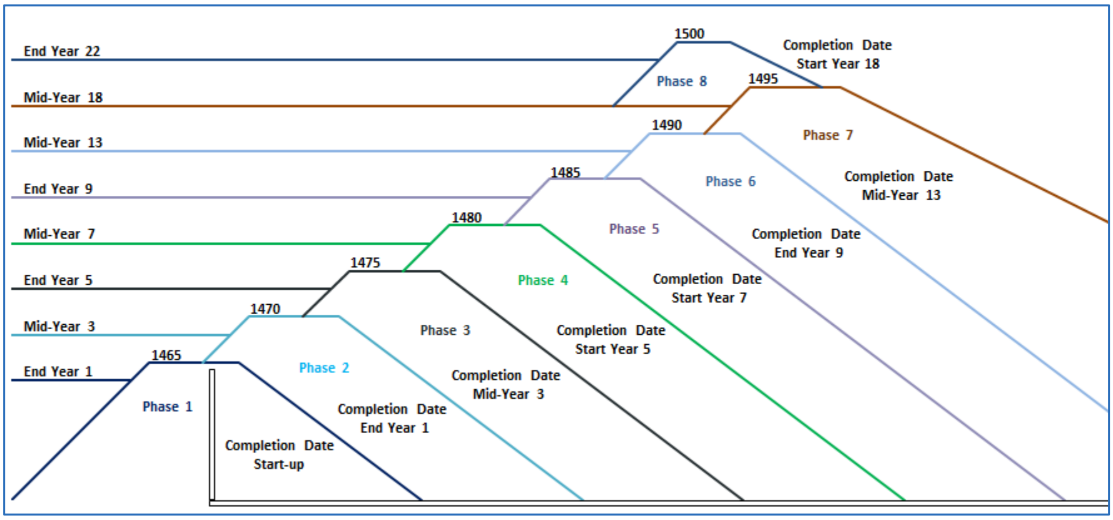


Figure by Epoch, 2017.

The final lift of the wall would be constructed as an upstream wall as shown. In order to confirm whether this is feasible, stability modelling of this option must be undertaken, as well as field investigations during operation of the facility.

18.4.9 TSF Construction Works

The construction of the TSF wall would include the following:

- Topsoil stripping to a depth of 300 mm beneath the TSF footprint.
- A box-cut to a depth of 500 mm beneath each wall impoundment.
- A compacted key below the Phase 1 wall impoundment which comprises the following:
  - Depth required shall be deep enough to remove the Kalahari sands layer.
  - 10.0 m wide base.
  - 1V:1.5H side slopes.
  - 3.5 m wide compacted bentonite-enriched earth layer to prevent excessive seepage under the wall.
- A compacted earth starter wall impoundment with the following dimensions.
  - 17 m high (i.e. crest elevation of 1465 m amsl).
  - 15.0 m crest width.
  - 1V:1.5H upstream side slope.
  - 1V:2H downstream side slope.
- A Curtain Drain inside the impoundment wall, to reduce the phreatic surface through the wall. This will comprise the following:
  - Starting at the base of the box-cut for each starter wall and ending 1.0 m below the top of the wall crest. This will comprise of filter material.
  - A 160 mm perforated pipe at the base of the curtain drain.
  - A 160 mm non-perforated outlet pipe, conveying water out of the wall.
  - A 300 mm non-perforated pipe to convey water to the RWD.
  - Manholes at each outlet pipe to monitor and control the drain flows.
- A stormwater run-off trench and berm around the TSF from which water is directed away from the TSF. The trapezoidal solution trench has the following dimensions.
  - 1.0 m deep.
  - 1.0 m wide.
  - 1V:1.5H side slopes.
- A stormwater diversion channel with its associated cut-to-fill berm with the following dimensions.
  - 1.0 m deep.
  - 1.0 m wide.
  - 1V:1.5H side slopes.

- A buried 900 ND Class 150D spigot-socket precast concrete penstock pipeline in each valley, composed of single intermediate intakes and a double final vertical 510 ND precast concrete penstock ring inlet;
- A 1500 micron liner along the bottom of each valley and approximately 200 m wide, in order to prevent tailings water seeping through the highly permeable Kalahari sands;
- A 280 ND slurry spigot pipeline along the length of the TSF perimeter; and
- A two-compartment reinforced concrete RWS.

The specified size of the penstock pipeline and the slurry delivery pipeline has been based on preliminary design calculations and should be re-evaluated during the next phase of the project.

As the impoundment walls will be constantly raised the side slopes have been set to 1V:2H external and 1V:1.5H internal. This both decreases the volume of the wall as well as the time required to construct each lift. The final Phase of the wall will have an external slope of 1V:3H. As no stability analyses have been conducted in the prefeasibility stage of this project, the configuration of the TSF side slopes will need to be re-evaluated in the next phase of the project.

18.4.10 TSF Depositional and Operational Methodology

The depositional technique selected for this project will be a valley impoundment, hydraulically deposited spigot facility. The impoundment wall will be constructed using waste rock or borrow material and tailings will be deposited behind the wall and into the valley. This design is a common construction technique used in tailings storage facilities. The three principal designs are downstream, upstream and centreline structures, which designate the direction in which the embankment crest moves in relation to the starter wall at the base of the embankment wall. The Kamoa TSF is a downstream structure. Figure 18.9 shows a simplistic diagram of the stages of construction of a downstream raised embankment.

The tailings are usually discharged from the top of the dam crest creating a beach and a resulting supernatant pool develops as far away from the wall as possible. Where the tailings properties are suitable, natural segregation of coarse material settles closest to the spigot and the fines furthest away.

Figure 18.9 Downstream Method of Embankment Construction

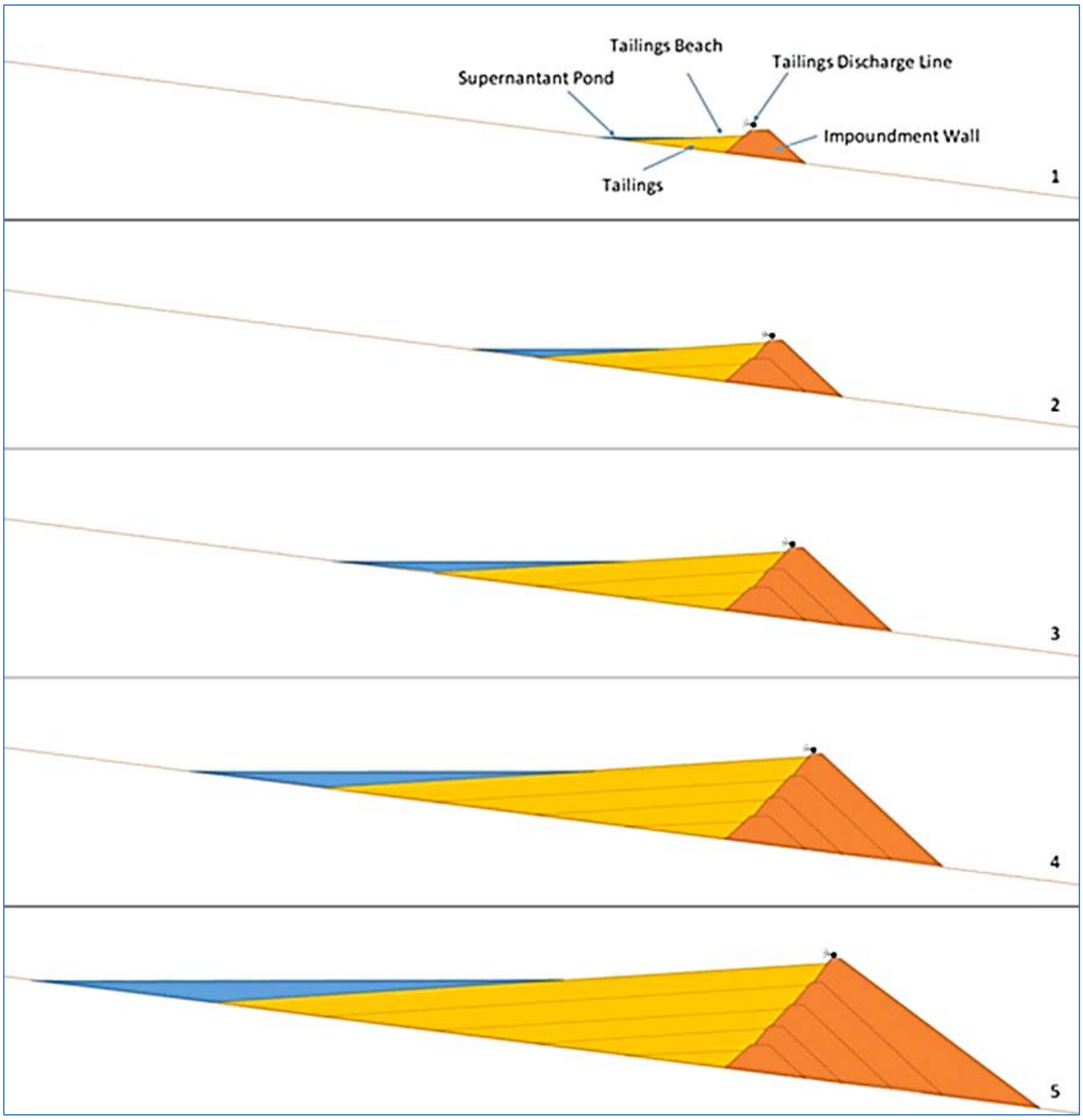


Figure by Epoch, 2017.

For the selected depositional methodology, tailings are deposited into the TSF basin via a spigot pipeline located on the inner crest of the perimeter wall as shown in Figure 18.10. During commissioning, deposition of the tailings behind the impoundment wall is directed to the base of the inner toe of the impoundment wall by flexible hoses. Deposition during this stage is to be carefully controlled, monitored and intensely managed to ensure that the wall is not eroded by the tailings stream.

Figure 18.10 Multiple Spigot Discharge

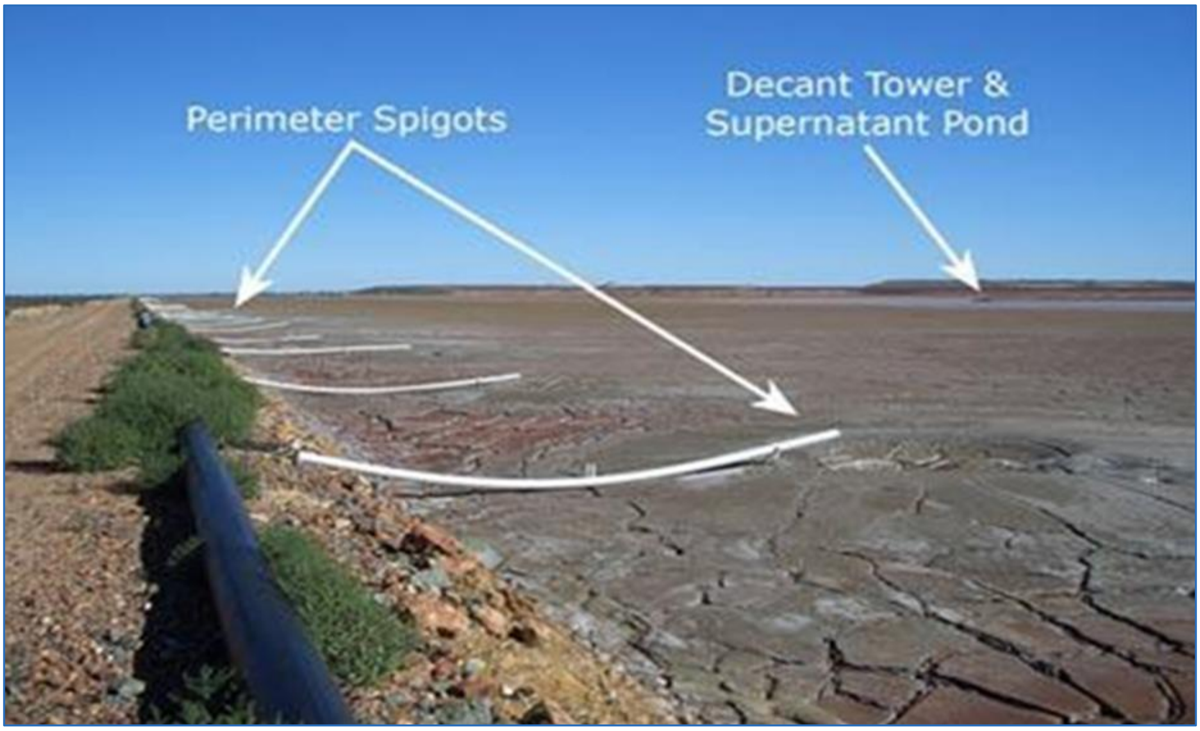


Figure by Epoch, 2017.

18.4.11 TSF Phasing

The TSF construction has been phased in order to delay some of the capital costs. The main construction items which have been phased are:

- The Impoundment wall and associated drains.
- The penstock.
- The HDPE liner.

These contribute the most to the cost of the project, thus they have been phased.

18.4.12 Water Balance

An overall water balance model for the TSF has not been undertaken during the PFS phase but should be considered during the next phase of the project. Based on Epoch's experience it is expected that the equivalent of 50 to 60% of the water in the tailings slurry will be recovered and returned to the plant as an annual average. A large amount of the tailings water will evaporate and be held in the dam with the settled solids, but rainfall will replace some of this lost water.

18.4.13 Closure Activities at Cessation of Operations

At the cessation of operation of the TSF, the focus will be on the cover and vegetation of the top surface of the facility, the decommissioning of facilities associated with the TSF and the construction of stormwater control measures as required. Specific activities that will be carried out will include:

- The dismantling and removal from site of all pipes and supports associated with the slurry delivery and return water systems.
- The decommissioning and plugging of all penstock inlets and outfall pipes.
- The construction of stormwater decant points to the TSF basin. The decant points will be located so as to control the rate of decant from the basin and will be constructed along the up-gradient side of the facility to minimise the flow velocities associated with the decanting process. The spillways will be designed to accommodate the peak design flows from the facility area and will be rock and/or concrete lined.
- The return of stripped soil from the footprint of the facility to enable the placement of a soil cover to the outer slopes and cover layer on top of the TSF.
- The placement of a mixture of soils and selected waste materials to the outer slopes of the impoundment and top of the TSF wall in preparation for the establishment of vegetation.
- The supply and hand planting of vegetation to the outer slopes of impoundment wall and top of the TSF to assist in the prevention of erosion;
- The aftercare and maintenance of the cover layers and vegetation.
- Minor earthworks to drains, roads, silt trap, trenches, etc.

The duration of the final closure process may be affected by the length of time required for the basin of the facility to dry sufficiently to enable the placement of cover material in preparation for the vegetation establishment.

18.4.14 Risks

The possible project risks associated with the current TSF design are as follows:

- Kamoa is situated in a seismically active area. No stability analyses have been undertaken for the TSF to confirm that the current TSF geometry will withstand a seismic event.
- A suitable borrow pit has not yet been identified for use in the impoundment wall.

18.4.15 TSF Recommendations

For the Definitive Feasibility Study stage of the project, it is recommended that the following be included:

- A more thorough geotechnical investigation of the TSF site in order to confirm the type, extent and characteristics of the in-situ materials as well as available construction materials.
- A more thorough water balance study for the TSF be undertaken.
- A seepage analysis and slope stability study be undertaken to confirm the seepage regime through the TSF as well as to confirm the TSF stability during a seismic event. The results of these analyses could impact greatly on the geometry of the TSF walls and ultimate height of the facility.
- Confirmation of the physical characteristics of the tailings product based on laboratory testing of a representative sample generated by the IFS4a flotation testing flowsheet. This must include flume and rheology tests to determine the tailings beach slope.
- An assessment of the need for additional contamination control measures such as HDPE liners or clay liners, dewatering and/or contaminated water treatment.
- Possible further optimisation of the TSF preparatory works in terms of layout, footprint extent, etc.
- Compilation of a more detailed schedule of quantities describing the proposed preparatory works and the pricing of the schedules to a greater level of accuracy.

18.5 Site Communications

Communication to the site is currently provided by high-bandwidth satellite internet connection provided by O3B with a Vodacom cellular data internet connection for back-up. Fibre optic internet service providers are operating in Lubumbashi 300km from Kolwezi and there are reasonable prospects for this to be extended to Kolwezi and Kamoa in the near future.

A fibre optic network has been installed across the site for the existing temporary facilities and this will be expanded as the permanent facilities are constructed. Cell phone coverage is available on site from Vodacom and Orange cellular providers. Radio systems are already operational at Kamoa and these will be expanded on surface and underground as the project is developed.

18.6 Site Waste Management

Currently land fill sites or waste collection facilities in the Kolwezi area are limited. There are hazardous waste management contractors or services based in Kolwezi that can deal with oils, batteries, bio hazardous waste etc. There are a number of companies collecting used oil for recycling and for use as burner fuel. Kamoa plans to construct a landfill site near the mine for non-hazardous waste disposal. A suitable site has been chosen for this and a concept design and costing for this has been prepared by Golder Associates.

An integrated approach to waste management for the project will be needed. This would involve reduction, reuse, recycling and would be done onsite through waste separation. Some of the methods incorporated would be through composting, alternative uses based on stockpiling areas and storage for other disposal (for hazardous chemicals like oils, batteries, vehicle filters and old parts etc.). This approach will be developed further during the feasibility phase.

18.7 Roads and Earthworks

18.7.1 Main Access Road

The current access road to site runs through Kolwezi town and is relatively long and in poor repair. A shorter, reliable and safe main access road from Kolwezi to the Kamoa-Kakula Project will be required. It is planned to construct a new road on the southern side of Kolwezi, avoiding driving through the town and connecting Kamoa to the N39 national road. Detailed design of the main access road is currently underway. The proposed new and upgraded road sections are shown in Figure 18.11.

Figure 18.11 Proposed Access to Kamoa Site

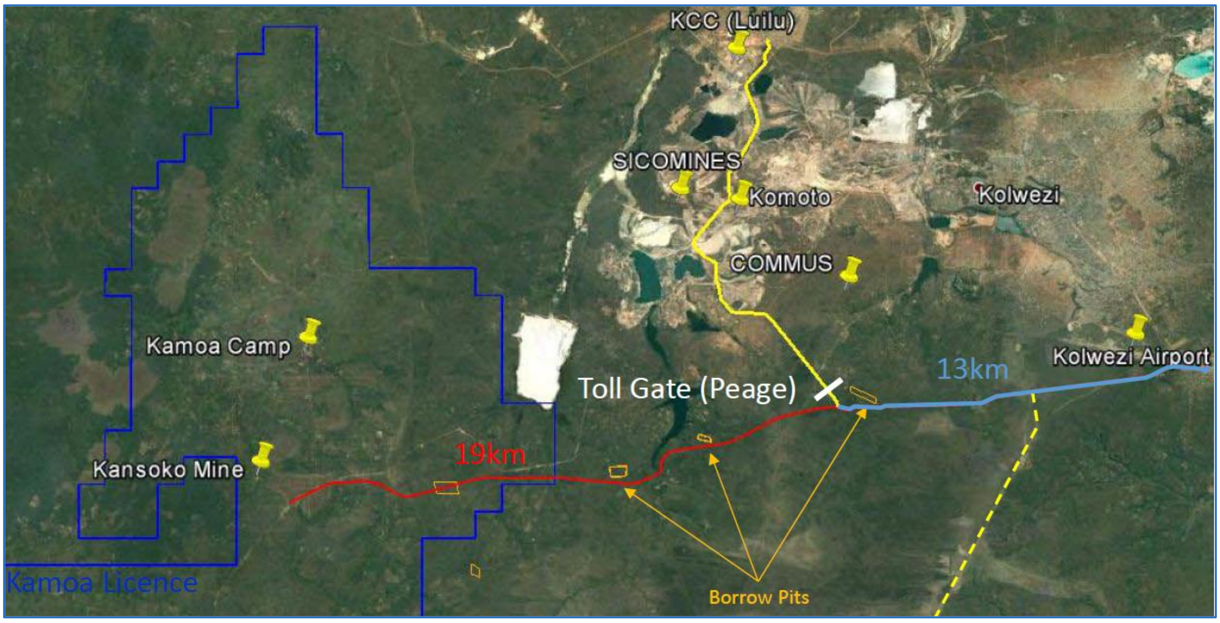


Figure by MDM, 2017.

The new road is to be gravel but will be built up substantially to achieve the necessary drainage. A cross-section of the road is shown in Figure 18.12.

Figure 18.12 Proposed Access Road Construction

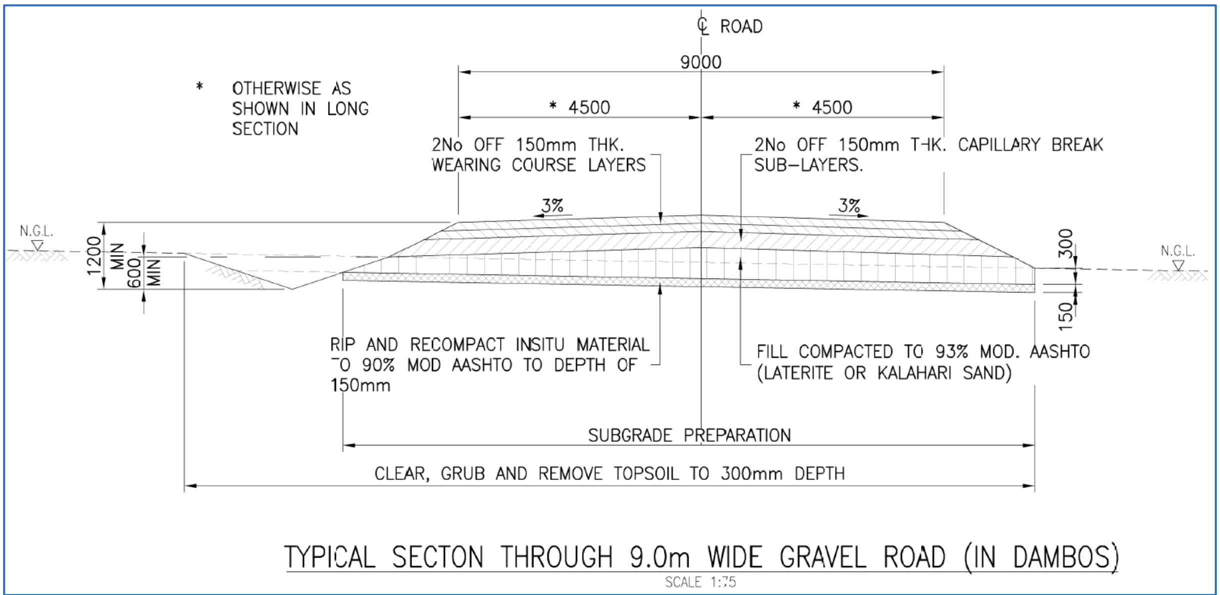


Figure by MDM, 2017.

The design of the gravel road was completed in such a manner to convert it to a bitumen surfaced road in the near future. The access road connects Kansoko Mine to the National Road N39, in the vicinity of Kolwezi Airport. The road comprises of approximately 31.8 km of gravel road.

The access road to the Kansoko Mine will have a 55 m minimum road reserve. Typically, the road will be designed for a vehicle speed of 80 km/h and operating speed may vary from 60 km/h to 80 km/h depending on the surrounding locations and constraints. The road will have a surface width of 9 m with two 3.5 m lanes and 1.0 m surfaced shoulders. The inclusion of gravel shoulders is mainly for the recovery area. The road includes 3 river crossings that will be constructed with steel culverts. The road base layers will be constructed from in-situ material adjacent to the road and the final layer works and wearing course would be constructed from laterite sourced from 3 or 4 borrow pits identified near to the road.

The permitting and land compensation processes have already started for this road and firm tenders have been obtained from 5 potential contractors for the construction. It is planned to construct this road as early as possible to provide safe and quick transport between Kolwezi and Kamoa during the construction period of the mine and plant.

18.7.2 Other Roads

The following facilities have been allowed for inside the plant and mine area:

- Plant roads. All plant roads will be gravel roads.
- Plant to portals roads. A 6 m wide gravel road will be provided.
- Plant to tailings storage facilities. A 6 m wide gravel road will be provided.
- Service roads (conveyor, ventilation fans, slurry pipelines). 4 m gravel roads will be provided as serviced roads.
- Village access road. A 6 m gravel road will be provided.
- Village roads. Varying road widths will be provided, depending on the hierarchy of the road in the village. All roads will be surfaced roads.

18.7.3 Terracing and Earthworks

Terracing shall be designed with suitable grading for efficient draining of stormwater run-off and keeping in mind optimisation of cut-and-fill earthworks quantities. Stepped terraces shall be proposed to accommodate mechanical and process requirements on the plant. The Kamoa site has been identified to consist of collapsible soils of low bearing capacity that will not provide adequate support for heavy structural foundation loads. Therefore, terrace layer works shall be designed for removal of unsuitable in-situ soil and backfilling with structural fill layers to provide a stable founding medium for structural foundations to carry heavy mechanical and process equipment. For major foundation loads such as the ball mills, piling will be required. All topsoil will be stripped from terrace areas and stockpiled for use during site rehabilitation.

18.8 Logistics

A phased logistics solution is proposed. Currently the North–South corridor between southern DRC and Durban or Richards Bay in South Africa is viewed as the most attractive and reliable export corridor. Product would be transported by truck to Ndola in Zambia and then loaded on to trains for onward transport to the ports of Durban or Richards Bay in South Africa.

Later it is planned to use the existing 2,000 km rail line between Kolwezi and the Angolan port of Lobito. This line has been re-built for 1,600 km between Lobito and the Angolan-DRC border at the town of Dilolo and can handle a capacity of 20 Mtpa. The 400 km on the DRC side of the border, from Dilolo to Kolwezi, is in a poor condition and needs major repair and upgrades. As soon as this section has been sufficiently rehabilitated and put into operation, Kamoa will construct a private 20 km rail spur linking the mine to the main line and product will be railed directly from the mine to Lobito for export. It has been assumed that this rail link will be available from Year 3 and reduced freight costs have been assumed from this time.

A number of alternate export corridors will remain available to Kamoa and could be used if necessary. Apart from the North–South corridor to Durban and the Lobito/Benguela corridor to the West, the Tazara corridor to Dar es Salaam in Tanzania and the option of exporting some volume through Walvis Bay in Namibia also exist.

Concentrate will be bagged at the mine and road hauled to the closest facility in Northern Zambia where freight can be transferred from road to rail. A number of road hauliers are active on this route. It has been assumed that a new intermodal (road to rail) facility will be available in Chingola, 45 km by road from the DRC/Zambia border at Kasumbalesa. Zambia Rail (ZRL) and a number of private logistics companies are considering developing more rail linked facilities further north of Kitwe, which is currently the northern most and closest rail linked facility to the border with the DRC. Bagged concentrate will then be packed into 20 ft containers at the port.

The western rail corridor to Lobito and the North–South corridor through to Zambia is shown geographically in Figure 18.13 and diagrammatically in Figure 18.14. The North–South Corridor is shown in diagrammatically in Figure 18.15.

The use of an operational line between Kolwezi and Lobito port is not exclusively dependent on the rehabilitation of the rail infrastructure. It needs joint agreement from both countries' respective governments, in addition to completing an institutional framework that should govern these operations. It also requires the DRC national rail authority (SNCC) to award a private concession to upgrade and operate the rail.

Figure 18.13 Kamoa to Lobito Rail System

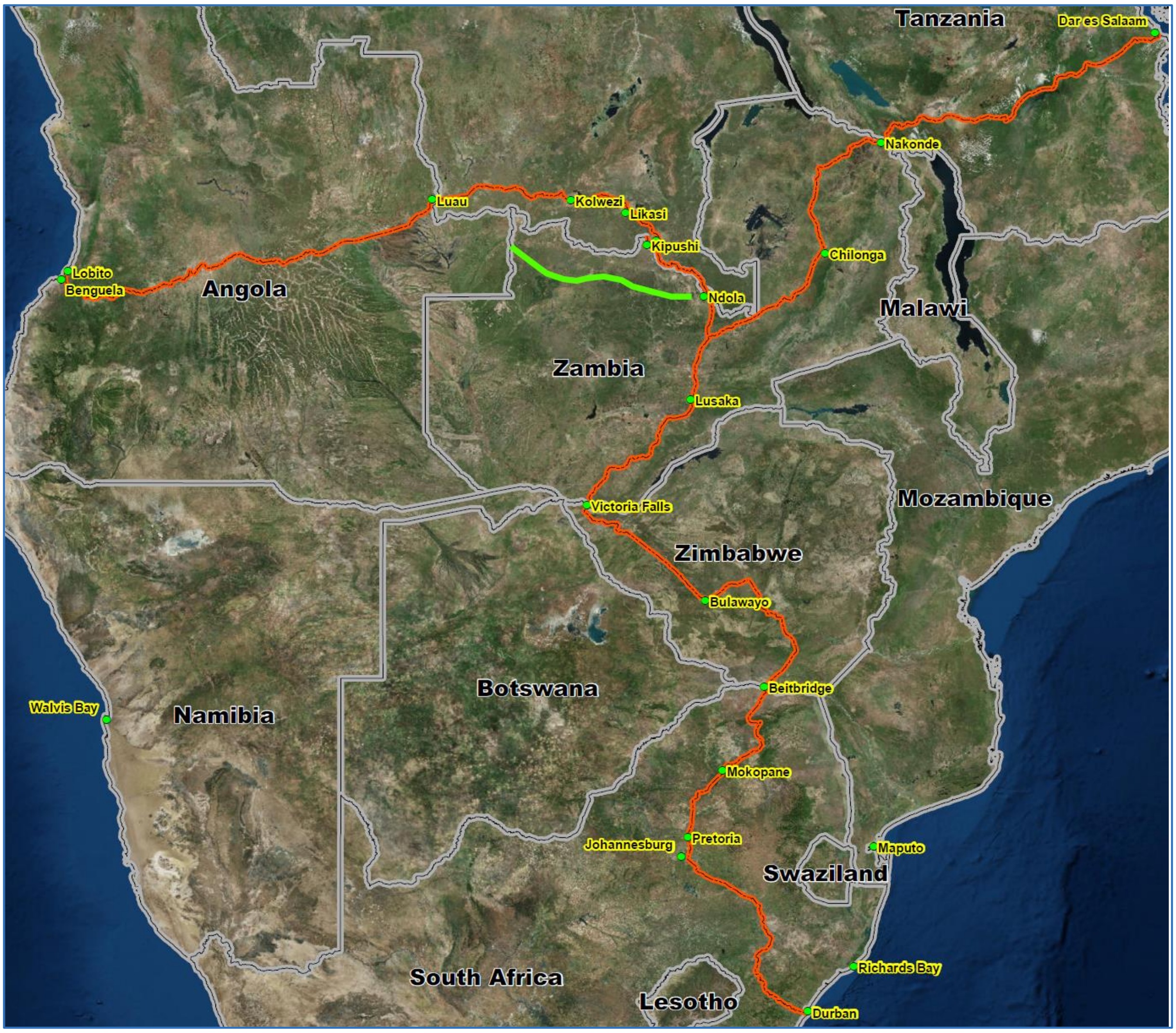


Figure by Grindrod, 2017.

Figure 18.14 Western Rail Corridor

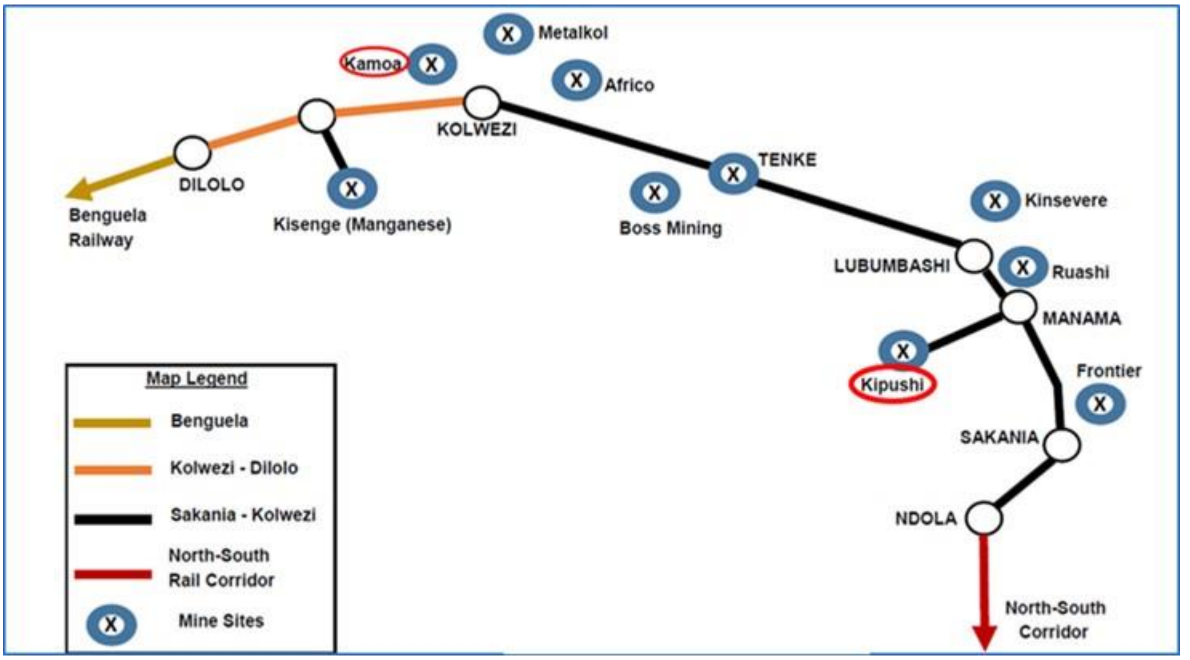


Figure by Grindrod, 2015.

Figure 18.15 DRC to South Africa North–South Rail Corridor

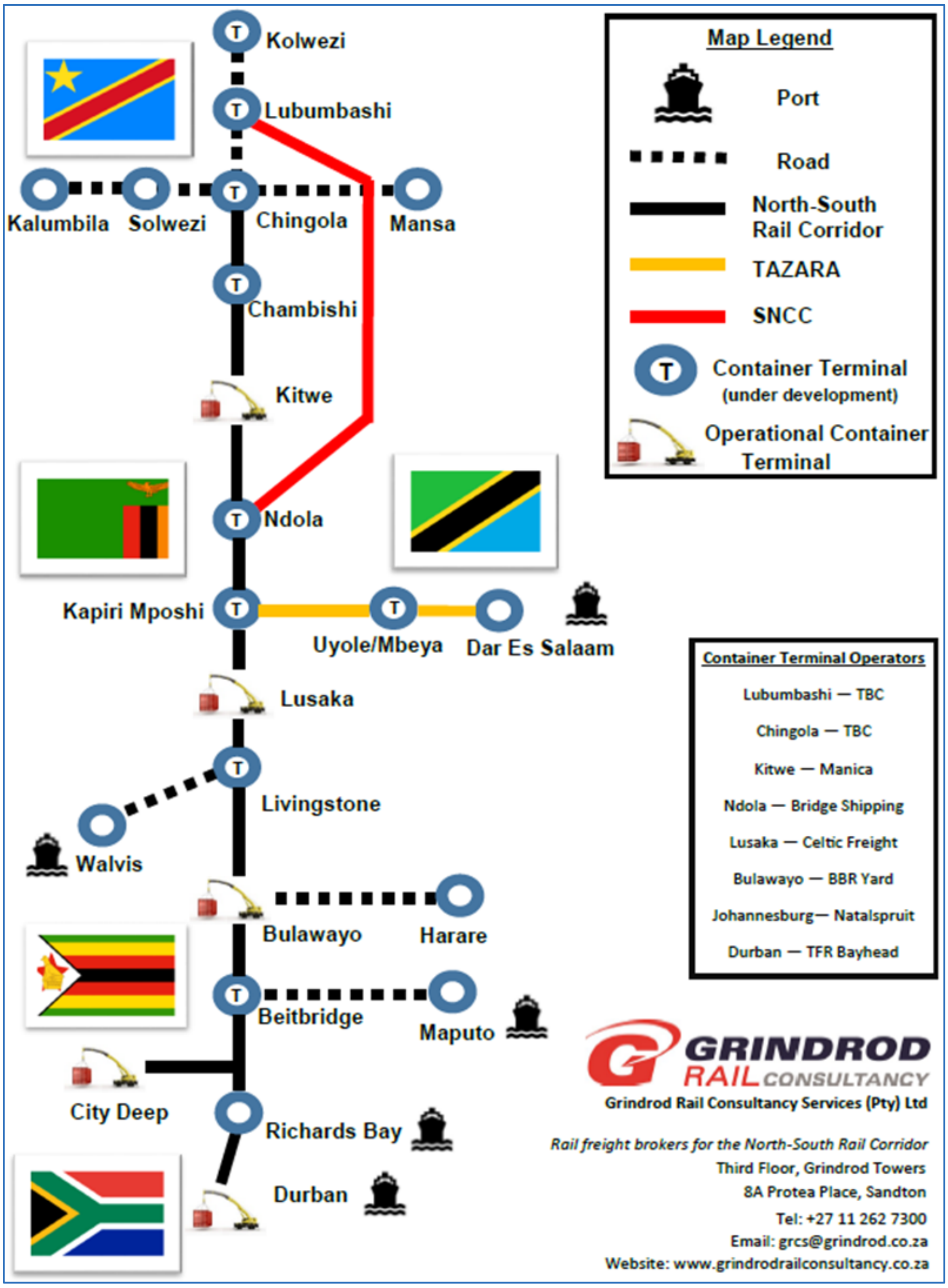


Figure by Grindrod, 2015.

18.9 Airports

Lubumbashi International Airport in DRC has an elevation of 1,197 m above mean sea level. It has one runway designated 07/25 with an asphalt surface measuring 3,203 m by 50 m. This airport is regularly serviced by the following airlines: South African Airways (operated by South African Express), ITAB (DRC domestic airline), Kenya Airways, Ethiopian Airlines, Congo Express, and a number of smaller airlines and private charters.

The Kolwezi airport is located about 6 km south of Kolwezi. The airport has an elevation of 1,526 m above mean sea level. It has one runway designated 11/29 with an asphalt surface measuring 1,750 m by 30 m. This airport is largely serviced by Air Fast, providing 4 flights a day between Lubumbashi and Kolwezi. There are plans by the Lualaba Provincial Government to upgrade Kolwezi airport to an international airport and to lengthen the runway to be suitable to receive aircraft from Europe and South Africa. It is currently possible to make special arrangements for charter flights to fly directly from Johannesburg to Kolwezi. When the Kamoa passenger numbers increase sufficiently during construction it is planned for Kamoa to operate such a service 2–3 times a week.

Kamoa is currently in the process of upgrading some office and waiting room facilities at Kolwezi airport and these premises will be rented from the Airport Authority for Kamoa's exclusive use. The airports will be utilised to transport people, goods and material to the project site during construction and operations phases.

18.10 Consumables and Services

18.10.1 Fuel

Transport fuel and fuelling infrastructure is available along all of the required routes to Kolwezi, albeit fuel quality and standards between countries are likely to vary. 50 ppm Sulphur fuel is readily available in the region. On site, it is planned for 2 or 3 fuel depots and filling stations to be owned by Kamoa and operated on a consignment basis by fuel suppliers. It is planned to have two weeks of fuel storage on site to enable uninterrupted operation during periods of delivery delays.

18.10.2 Maintenance

Workshops facilities will be constructed at Kamoa for activities including vehicle repairs and major overhauls, a boiler making shop and a machine shop. Kamoa needs to be relatively self-sufficient in terms of workshop facilities. However, there are some major workshop facilities operated by Gecamines in Likasi which could be used for some major machining work, as well as various smaller general workshops in Lubumbashi, Likasi, and Kolwezi.

18.10.3 Inbound Project Logistics

The provision of logistics services should be structured in a way that will best negate the risk associated with transport and freight forwarding for the project. To achieve this, a primary freight forwarding contractor should be appointed for the international component of the route. A secondary partner should be considered, to assist with supply from South Africa and other over flow requirements, if required. A local DRC customs clearing/broker partnership should also be established. It should further be ensured that the applicable protocols are implemented to allow goods to move on a duty-free basis between countries of supply and/or transit. Central warehousing facilities should be set up, to consolidate transport loads and to ensure that bonds are not retained on shipping containers. A bonded area on site has been allowed for.

There are no major road restrictions in terms of load sizes and masses for transporting equipment to site. The 2 bridges between Lubumbashi and Kolwezi that were a restriction in the past have been upgraded to carry abnormal loads.

Currently freight from South Africa to Kamoa takes about 3 weeks, including customs clearing. During construction, it will be critical to implement an efficient logistics process flow, expediting and tracking system to avoid construction delays.

18.10.4 Operational Inbound Logistics – Reagents and Consumables

During the operational phase, reagents and consumables should be sourced and transported from South Africa, unless suitable reagents and/or consumables can be sourced in the DRC and/or in neighbouring countries. Reagents and consumables procured from outside of South Africa should be moved through Durban harbour, unless other routes are found to be more economical. For suitable reagents and consumables, the railway should be considered as an option.

18.11 Water and Wastewater Systems

18.11.1 Water Demand

The estimated water demand for the project scenario is given in Table 18.4. These figures are an average through the year. There will be a large variation between dry and wet seasons. A contingency has been added to account for unanticipated consumption, such as increased tailings dam water retention due to finer tailings P_{80}.

OreWin

Table 18.4 Estimated Water Demand

Description	Units	Quantity
Mining Water Requirement	m^3/day	320
Concentrator Water Requirement	m^3/day	7,600
Potable Water Requirement	m^3/day	280
Contingency	%	10
Total Daily Requirement	m^3/day	9,100

Raw water will be provided to the site via the four production boreholes forming the Southern Wellfield, as identified by Kamoa. The boreholes will be connected to a common overland pipeline (7 km) which will feed into a water storage dam located at the plant. This will provide all necessary raw water which will then be used to provide the required process water makeup, gland water, fire and reagent make-up water. Most water loss is due to evaporation and seepage from the TSF. It is estimated that the equivalent of 50%–60% of water going to the TSF will be returned. A return water pipeline (10 km) will bring water from the TSF to the process water tank. Water from mine de-watering will also be utilised for process water make-up.

18.11.2 Bulk Water

The assessment of the bulk water supplies has been undertaken with the view of supplying the estimated water demand of 9.1 ML/d.

Two potential sources have been identified for the bulk water supply. The first is the aquifer within the sandstone forming the Kamoa and Makalu Domes, and also constitutes the footwall to the mining operations. The second potential source is the major rivers within the Kamoa Mining Licence, including the Lulua, Tjimbudgi and Lufupa rivers. The rivers have strong flow year-round and sufficient water could be extracted with a simple weir arrangement.

River water is considered a contingency at this stage, since it is estimated that sufficient water will be available from boreholes and mine de-watering. The bulk water supply will be obtained from the 4 boreholes (3 production and 1 standby holes) forming the Southern Wellfield. This supply will be augmented by water obtained from the decline dewatering boreholes.

The bulk water supply could be augmented by groundwater inflow into the underground workings. The volume of mine water inflow will be determined in the future.

According to the DRC Mining Code, a mining licence gives the holder automatic rights to use the surface and ground water on the licence area, so there is minimal permitting risk for use of this water.

18.11.3 Potable Water

Potable water for mining, ablution facilities, kitchens and emergency stations (eyewash and showers) will be obtained from boreholes and treated by means of disinfection only (chlorination). An appropriate drinking water standard will be applied, referencing indicators such as bacterial content, residual chlorine, turbidity, and dissolved solids. The borehole water at Kamoa is very good quality, with exceptionally low dissolved solids levels.

Potable water will be distributed via pipe racks and sleeper ways along with other services where possible and underground as necessary.

18.11.4 Stormwater Infrastructure

The Department in Charge of the Protection of the Mining Environment in the DRC requires that an Environmental Impact Study (EIS) is performed for any proposed mining activity within the DRC. The EIS is prepared using the Mining Regulations, Annex IX (Walmsley, B. & Tshipala K.E., 2012). Article 19 of Annexure IX requires that all mines develop measures to reduce the inflow of uncontaminated run-off water into the mining site water management system. Article 82 of Annexure IX requires that the sizing of any water retention structures accommodates for the water contribution resulting from a projected 24-hour flood with a return period of 100 years. The sizing of the stormwater management plan, the pollution control dams and the pipelines with their required pumps are all based on these regulations.

18.11.5 Stormwater Management Plan

The assumptions made for this investigation include:

- Due to the lack of sufficient data closer to the Kamoa site, the Solwezi rainfall data was used to analyse the one in 100-year return period 24-hour rainfall event.

The stormwater management plan and pipeline system were developed based on the most current site arrangement information available to Golder Associates Africa.

The location of the plant, stockpile and decline area is shown in Figure 18.16.

The run-off from this area will be contained with earth dams and will need to be managed within the mine's dirty water system. Berms are required around the perimeter of the area to prevent run-off from the upslope areas entering the site. The run-off from the site is collected in berms/channels located on the northern perimeter of the area. The run-off collected by these berms is directed to a stormwater control dam located to the north of the site. The capacity of the stormwater control dam is sized to store the run-off volume from the 100-year 24-hour storm event.

The 1:100-year 24-hour storm depth of 139 mm, calculated using the daily rainfall data measured at the Solwezi rain gauge, was used to calculate the run-off volume that would report to the stormwater control dam. The run-off from the catchment for the 100-year event will not be 100%. There will be losses both from depression storage and infiltration. The SCS technique was therefore applied to calculate run-off from this event. Based on a catchment area of 66 ha, a flood volume of approximately 58,000 m^3 was estimated for the 100-year 24-hour event. This capacity is therefore recommended for the stormwater control dam.

The area of the stormwater dam is 1.5 ha, with a 4 m depth. The dam is assumed to be a cut-and-fill dam with the wall material sourced from the dam basin. Geotechnical studies will be required to confirm the suitability of the materials for dam construction. The required lining for the dam will be determined during the EIA, but allowance for a liner in the costing is included at prefeasibility stage.

The stormwater management plan included in this document is done at a high level and should be considered a conceptual plan. A more detailed stormwater management plan and pipeline system will be developed as the mining project progresses.

Figure 18.16 Stormwater Dam

Figure by MDM, 2017.

18.11.6 Wastewater

Sewage from kitchens, laundries, and ablutions will drain via underground sewers to a sewage treatment plant and will be treated to produce an effluent of a suitably safe standard for process use.

Floor washings that contain organic contaminants, from kitchens and ablution blocks, will also drain via the sewers to the treatment plant. Floor washings that are potentially contaminated with mineral oils (workshops, refuelling and lube and diesel storage areas) will drain to the run-off dam.

Kamoa currently has a sewerage plant for the existing accommodation camp and similar plants will be utilised at the mine site and future accommodation camp. These plants are zero-sludge plants, fully digesting solids into solution. The treated water would be used for irrigating gardens or be recycled to the concentrator process plant. Other wastewater streams and by-products such as acid are covered under plant process design.

18.11.7 Potential Water Treatment

It is predicted that during the initial stages of mining, all excess water will be re-used at the plant as make-up.

However, as mining progresses with bigger voids forming, larger volumes of ground water could be expected within the underground workings, which will require dewatering.

The mine water is not expected to be acidic. Initial treatment will largely involve settlement, removal of oil and grease, etc. High concentrations of nitrate may also have to be removed as well as any heavy metals.

However, as the water balance shifts to positive over the life-of-mine, including seasonal fluctuations, the acidity of the water could increase, necessitating treatment by installation of a water treatment plant.

A high-level capital cost estimate for a 1 ML/day plant to address acidity, presence of metals and salts in the mine water will amount to approximately US$1 M.

The cost for a water treatment plant could be either provided for through the contingency provision for the project or from the closure cost provision for the Mine, especially in the event that water treatment is required beyond closure.

An option for treatment of excess waste water is to use evaporative mist sprays over the TSF. Much of the water evaporates from the mist and the remaining water containing dissolved salts and solids falls into the TSF. This method is successfully being used at a number of other mines in the region. However, it can only be used during windless and sunny periods.

18.12 Fire Protection and Detection

The fire protection and detection system for the surface plant and infrastructure (excluding all underground mining which is covered separately) will be developed in consultation with and subject to final approval from the Owner's risk assessors. The system will be designed to comply with DRC legislation (where applicable), the project Health and Safety standard/s, project specifications and fire protection standards as adopted by the Project.

The development of the fire protection and detection system will take into account all high-risk areas of the plant, as these may require specialised fire systems. The system will include a combination of passive measures (e.g. fire walls, physical isolation etc.) and active systems (e.g. fire detection, fire water systems, gas suppression systems, etc.).

Fire detection equipment will include a Fire Indicator Panel (FIP) located in the main control room area, and local intelligent Sub Fire Indicator Panels (SFIP) as required located around the site.

The fire detection system will be independent of the Process Control System (PCS) and will be specified as part of the overall Fire Protection System for the plant, which will also include the Fire Water System, Gas Suppression Systems and any other specialised systems (if required for high risk areas).

Fire water storage will be a dedicated water supply volume, sized in accordance with the requirements of the applicable fire standard. The fire water pump house will be designed with a high degree of reliability, and would typically include a jockey pump (to maintain system pressure under normal non-fire conditions), as well as electric and back-up diesel fire water pumps.

The water supply will be sized to provide the required maximum firewater flows for any single fire event. Fire water will be distributed around the plant via a fire water reticulation network, which will connect to strategically placed hydrants, hose reels, sprinkler systems, deluge systems, and / or foam systems as required.

Buildings and offices will be equipped with hose reels and portable extinguishers, in accordance with the governing building standards and project specifications.

Gas suppression systems will typically be used for critical areas such as electrical rooms, control rooms, server rooms etc. Hand-held extinguishers will be distributed around the plant and in all buildings.

The size of the site will require the availability of at least one fire fighting vehicle (with 4 x 4 capabilities) to ensure it is available to deal with fire events in remote areas of the site.

18.13 Hospital and Medical Facilities

The clinic and first-aid facility will be housed together at a suitable position near the main gate. The clinic will be suitable for all occupational health checks, regular consulting rooms, emergency trauma rooms, and 8 hospital ward beds for overnight patients. A separate first aid facility at the mining control room has been included for dealing with mine accidents. Medical equipment, including an ambulance, will be provided. Medical evacuation for ex-patriot employees will be provided by an outside contracting service. Hospitals are available in Kolwezi for employees and their families resident in Kolwezi.

18.14 General Building Requirements

The surface building requirements were obtained from other projects with the similar number of personnel, fleet size and production rates. An all-inclusive rate per square metre of floor area was applied for steel and brick structures. The estimate includes furniture, fitting, electrical appliances, power supply infrastructure and communication. The estimate is based on contractor construction. It is planned to erect some of these buildings early in the construction period so they can be used during construction, thereby minimising the requirement for temporary construction buildings. The buildings are described in the following sections.

18.14.1 Concentrator Buildings

- Administration building/offices (522 m^2).
- Clinic and first aid station (600 m^2).
- Kitchen /canteen (442 m^2).
- Change house (344 m^2).
- Weighbridge control room (22 m^2).
- Laboratory (960 m^2).
- Gatehouse and security (277 m^2).
- Training Centre (1,200m^2).
- Mess Complex (450 m^2).
- Satellite ablutions (25 m^2).
- Substations.
- Explosives storage (bunding allowance for mining explosives handling).
- Plant and Vehicle workshop (light crane loads) (433 m^2 and 300 m^2).
- Concentrate Handling (5,720 m^2).
- Plant stores (933 m^2).
- Plant control room (126 m^2).
- Reagents store (672 m^2).

18.14.2 Mine Surface Buildings

- Aggregate and multipurpose store (281 m^2).
- Briefing Area (400 m^2).
- Capital store (536 m^2).
- Change House Complex (3,801 m^2).
- Engineering Workshops (719 m^2).
- Firewater pump station (43 m^2).
- Medical room (77 m^2).
- Mine Rescue Room (77 m^2).
- Shaft Control Room (389 m^2).
- Shaft Gate House (275 m^2).
- Shaft Offices (1,419 m^2).
- Surface gas store (70 m^2).
- Surface lubricant store (39 m^2).
- Surface paint store (39 m^2).
- Tyre store (434 m^2).
- Warehouses (Stores) (1,417 m^2).

18.15 Owner's Camp

18.15.1 Accommodation

A permanent village will be constructed at the existing exploration camp location to provide accommodation for owner's team management, expatriates and EPCM consultants. Single units will be two bed with en-suite bathroom and family/executive units with 3 bed, two bathrooms with open plan living room and kitchen.

18.15.2 Facilities

The following facilities will be included:

- New kitchen and mess complex.
- Recreation centre.
- Sports facilities.
- Administration offices.
- First aid room.
- Laundry.

Wireless internet and cable TV in all rooms will be allowed for.

18.15.3 Roads and Services

The following roads and services will be provided in the accommodation area:

- Perimeter security fence.
- Gravel access roads to housing units.
- Parking (remote from rooms).
- Water reticulation, sized for fire flows and provided with hydrants.
- Sewer reticulation and treatment.
- Internal communications.

18.16 Construction Facilities

To facilitate the execution of the project, various temporary facilities are required. These facilities include:

- Construction Camp: A 1500 bed construction camp to accommodate the construction workers during execution will be erected within walking distance of the mine. The camp plan assumes single-quarters accommodation and will include bedrooms, ablution facilities, dining area and kitchen, recreation area as well as admin offices and guard house. Services such as water, sewer, electricity, in-room wireless internet and TV will be provided. As the camp will be used during the operational phase for on-shift accommodation it will be built for a 25-year service life. All ablutions will be shared. Seniors will each have their own rooms and Juniors will be two per room.
- Construction Site Offices: The Concentrator and Mining infrastructure buildings will be erected to be utilised as construction site offices. Once construction has been completed, the buildings will be refurbished and handed over to operations.
- Laydown areas: Contractors will need prepared areas to establish their site offices and areas to store construction material, equipment and vehicles. Fenced terrace areas with water, sewer and electrical connections will be provided.
- Customs Clearance Area: To facilitate the smooth delivery and release of construction material ordered from outside the DRC, a customs clearance area (bonded) will be created on site from which a customs clearance official will check, register and release all imported construction material. Fenced terrace areas with a storage shed have been allowed for.

18.17 Comments on Section 18

Infrastructure planning was completed at an appropriate level of accuracy for the Kamoa 2017 PFS and no issues were identified that will have a material negative impact upon the financial viability of the project.

The finer particle size for tailings will have an influence on water demand and more testing is needed to quantify the impact. A contingency has been included to allow for increased raw water demand.

19 MARKET STUDIES AND CONTRACTS

The Kamoa 2017 PFS assumes that copper concentrate will be sold at industry standard terms. The current market outlook is for a long-term concentrate treatment charge of $80/dmt concentrate and refining charge of 8 cents per pound of copper. This has been used in the economic analysis for the Mineral Reserve. The following is the copper payable scale for the various grades of copper concentrate; <30% deduct 1.0 unit, <33% deduct 1.1 units, <36% deduct 1.2 units, <40% deduct 1.3 units and >40% deduct 1.4 units. The base case analysis for the Kamoa 2017 PFS assumes a copper price of $3.00/lb, this is consistent with long term estimates and pricing used in other published studies. The economic analysis has allowed for a transport cost assuming all concentrate is treated in China.

There is potential to sell copper concentrate to smelters in Zambia and or merchants where more favourable terms may be possible. The potential sources for concentrate sale in Zambia are:

- Mopani Copper Mines (MCM) – Mufulira Copper Smelter.
- Chambishi Copper Smelter Limited.
- Konkola Copper Mines plc.
- First Quantum Kansanshi Smelter.

Mopani Copper Mines (MCM) operates the Mufulira Copper Smelter. MCM is majority owned by Glencore International and First Quantum Minerals Ltd holds a minority interest. The MCM smelter (ISASMELT) has a nominal smelting capacity of 300 ktpa copper. They do not produce enough concentrate from their own mines and purchase or toll concentrate from third parties.

Chambishi Copper Smelter Limited (CCS) is owned 60% by Yunnan Copper and 40% by China Nonferrous Metal Mining Company (CNMC) and the smelter began operation in 2009. The smelter is located about 30 km east of Chingola. CCS produce blister copper; they do not have a refinery. Their feed grade ranges between 28% and 48% copper with an average target of 32%. Historically approximately 50% of the concentrate feed is produced from their mine and the balance is purchased from Barrick (Lumwana), First Quantum (Kansanshi) and other small mines in the area. The blister is shipped to various locations and customers in China, Korea, Germany and India.

Konkola Copper Mines plc (KCM) is a subsidiary of Vendanta Resources which owns 79.4% of the outstanding shares. The remaining 20.6% is held by ZCCM-IH, a Lusaka and Euronext listed company that is 87.6% owned by the Zambian Government and 12.4% by public shareholders. The nominal smelting capacity is 300 ktpa using the OUTOTEC technology. KCM's own mines produce about 50% of their feed and the balance is purchased from other companies. The final product (blister and cathode) is shipped to their rod plant in Dubai and to customers in China.

First Quantum has an ISASMELT smelter with a capacity of 300 ktpa copper at Kansanshi that is fed by copper concentrate feed from the mine.

19.1 Supply and Demand

19.1.1 Supply

Copper output from currently operating mines is reported likely to decline over the next decade, following the trend observed in the previous decade. This forecast decline is based on declining grades mined from the large porphyry resources that provide the bulk of current copper supply (Figure 19.1).

Figure 19.1 Copper Production Grade

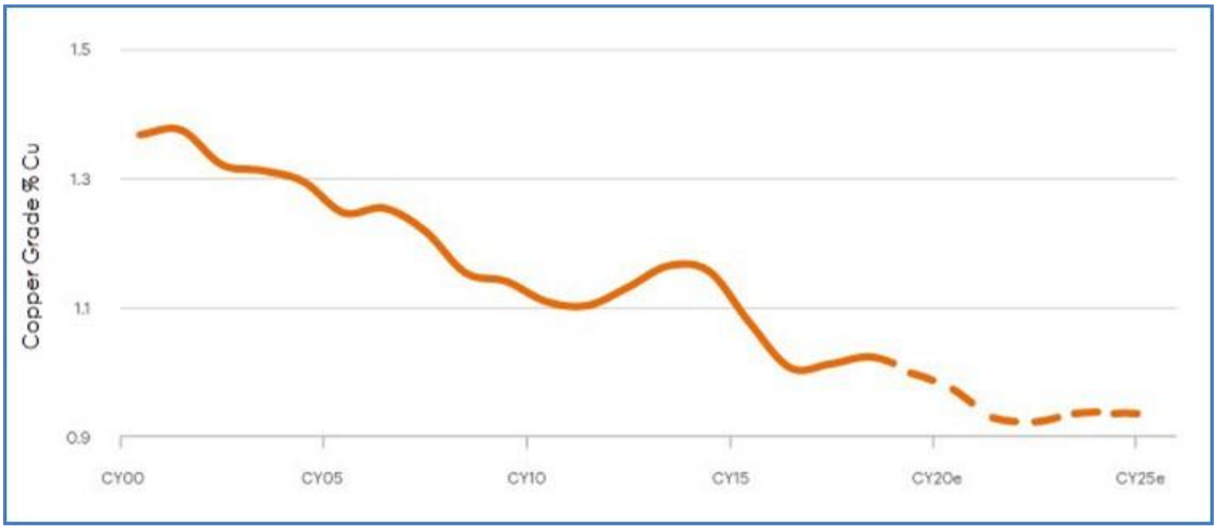


Figure by Wood Mackenzie, 2017.

To offset this grade drop and to meet the anticipated increase in demand, new mines will need to be developed. These mines will likely be higher cost, based on a range of macroeconomic, socio-political and geographic factors.

In the last decade, the rate of discovery of major copper deposit discoveries of similar scale to those that have historically under-pinned global supply has declined. Recent discoveries are focused in Africa with minimal equivalent major copper deposits discovered in other regions of the world. This slowdown in development of the very large deposits makes it extremely likely that the copper cost curve will steepen in the coming decade as generally lower quality and higher cost producers are brought on to fill the supply shortfall.

During a period of low commodity prices in recent years, major players in the copper industry under-invested in exploration and new projects, which is expected to result in constrained copper supply in the medium-term. In addition, exploration success has moved from OECD countries where the development risk is relatively low to regions of the world where development risk is relatively high, further curbing exploration activity and reducing the probability of bringing a deposit to development.

19.1.2 Demand

Future copper demand is expected to continue to be driven by on-going urban development and growth in household consumption, particularly in China and India, coupled with a world-wide move to lower emission technology.

Copper demand increases are directly linked to increasing urbanisation and improvement of living standards associated with economic development. China has entered this stage but still has greater than 50% of the population living in rural areas and a large proportion of the urban population subject to significant upgrade in household consumer goods and services. India lags behind China in terms of urbanisation and the level of household consumption for the urban based population. Other populous regions of South East Asia are expected to follow the growth trends observed in China and projected in India.

Major growth in the demand for lower emission technology such electric vehicles and renewable power is projected to drive increased future copper consumption.

Hybrid and full electric vehicles contain, respectively, twice and four times the copper content of conventional internal combustion vehicles. Both wind and solar power generation capacity are each more copper intensive than plants that rely on non-renewable energy sources. Relative to coal-fired plants, solar power generation facilities require more than twice the copper per KW while wind powered generation requires about five times the copper per KW.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Studies and Issues

20.1.1 Background

The mine licence is located in a rural area with no mining or industrial activities. Extensive urbanisation, industry, and mining occur in the nearby town of Kolwezi, 25 km to the east.

Kamoa Copper SA submitted a revised Environmental Impact Study (EIS) in January 2017 and received unqualified approval in March 2017.

The Project area is characterised by scattered, undeveloped rural villages and hamlets divided between the two groupings of Mwilu and Musokantanda. A total of 32 villages fall within the mine licence area. The population in the area has been recorded as 4,311 people (Golder, 2014), indicating a population density of approximately 10 people per square kilometre. The health services in the Project area are poor. Common diseases include malaria, tuberculosis (TB) and HIV which is often associated with HIV/AIDS infections.

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

Topographically, the Project area is at the edge of a north-north-east to south-south-west trending ridge which is incised by numerous streams and rivers. The elevation of the Project area ranges from 1,300 to 1,540 m amsl. Current exploration activities are at elevations ranging from 1,450 to 1,540 masl. The local topography of the Project is affected by the drainage catchments of the Mukanga, Kamoa, Tshimbunji, Lufupa and Lulua Rivers and the Kalundu, Kansoko, and Kabulo Streams (Figure 20.1). Recorded water quality indicates that ground and surface water quality is generally good, well within the DRC and international (World Bank) guideline limits although natural copper concentrations exceed these limits.

Ecologically, the Project area lies within the Central Zambezian Miombo Ecoregion. This ecoregion covers a large area, stretching north-east from Angola including the south-east section of the Democratic Republic of the Congo, the northern half of Zambia, a large section of western Tanzania, southern Burundi and northern and western Malawi. The climate is tropical, with a long dry season, up to seven months, which leaves the forest vulnerable to fires, and a rainy season from October to March. The woodland is interspersed with Dambos, (grassy wetlands), which may constitute up to 30% of the region. The woodlands in the study area contain typical Miombo flora of high trees with a poorly defined shrub layer. Typically, it has more evergreen trees than most Miombo woodlands. Approximately 50%–75% of the study area is currently considered to be degraded due to agriculture and charcoal production. Sensitive habitats include shrublands, Dilungus (large flat grassland areas forming the watersheds of most of the streams and rivers in the study area. They are comprised of extensive and deep sandy soils and act as sponges retaining rainfall and releasing water into the local drainage during the dry season), Dambos (valley bottom wetland areas) and the Miombo forest to the east of the Project area (Figure 20.2).

Radiation surveys carried out indicated that radiation levels are comparable with global background levels. As the natural background levels are not elevated these do not pose an increased radiation risk to the public.

20.1.2 Summary of Environmental Studies Conducted

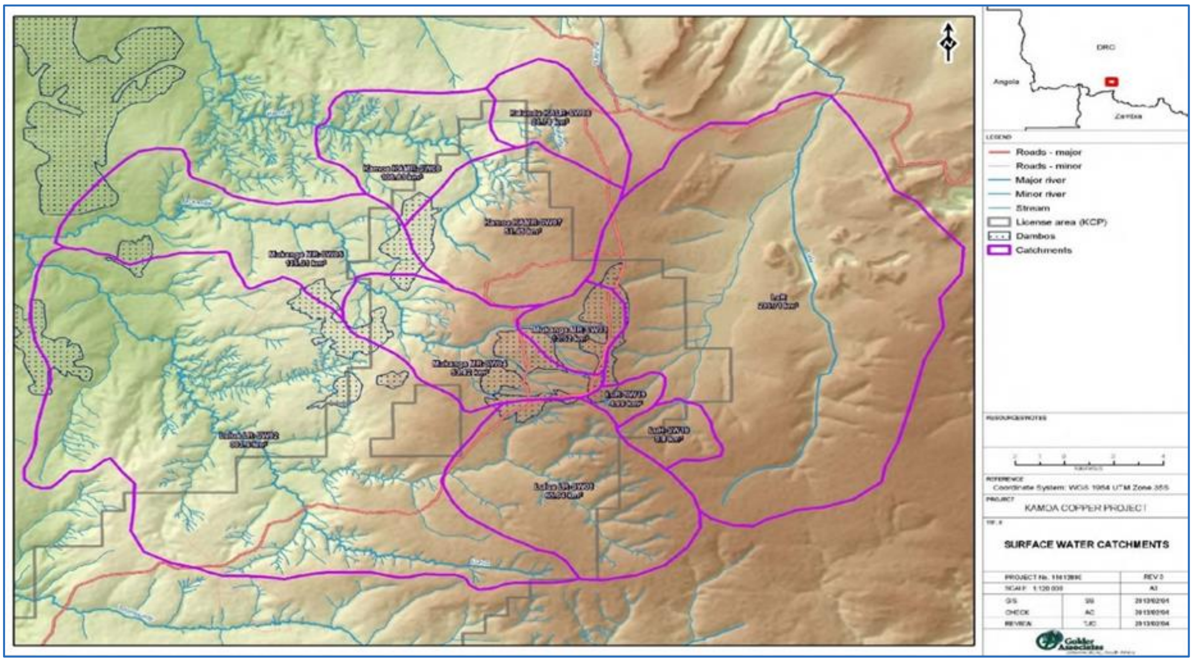
After carrying out exploration from 2006 to 2011, Kamoa Copper SA (Kamoa) (then known as African Minerals (Barbados) Limited (AMBL)) made an application to the Government to start mining in 2011. Authorisation to mine (called an exploitation licence) was given in August 2012. The application submitted by AMBL, included a description of the proposed Project (initial feasibility study) and an Environmental Impact Study (EIS) as required by DRC mining and environmental regulations, specifically - the Mining Code (Law No. 007/2002 of 11 July 2002) and the Mining Regulations, (Decree No. 038/2003 of 26 March 2003). The EIS provided an evaluation of environmental and social impacts of the Project and provided a list of actions the Project would implement to reduce the impacts and enhance or improve the benefits of the Project.

The EIS (African Mining Consultants, 2011) presented a provisional mining plan comprising of an underground copper mine for exploiting vast tonnages of high-grade ore through room-and-pillar mining, with surface processing to produce copper concentrate. Preliminary mine infrastructure locations were presented in the EIS. These included the locations of the Tailings Storage Facility (TSF) and supporting infrastructure such as employee accommodation, stores, access road and power supply. This EIS was based on conceptual planning information. This has subsequently been updated through the ongoing studies which continued since 2012.

Following the completion of the initial EIS, Kamoa Copper SA continued exploration activities and discovered additional resources. Further engineering and mining evaluation through the ongoing studies resulted in changes to the configuration of the Project. This initially involved the addition of a smelter (the addition of the smelter was subsequently deferred to Phase 2 and removed from the scope of the EIS update) and changes to the location of infrastructure, requiring an update of the EIS. It was further intended to upgrade the EIS to International Best Practices. Kamoa appointed Golder Associates (international engineering and environmental consultants (DRC registered)) in June 2012 to update the EIS, taking account of Project changes, to DRC legal requirements and to international standards. The update of the EIS, apart from updating the EIS itself, also included the following:

- Improved Project information.
- Plan of study (the Terms or Reference (ToR) to update the EIS informed by detailed scoping.
- Environmental, social and health studies (inclusive of ongoing monitoring).
- Ongoing community, interested and affected party as well as Government consultation.

Figure 20.1 Surface Water Catchments



Kamoa Copper Project Environmental Impact Study Update, Golder, 2017. Figure by Golder, 2017.

OreWin

Figure 20.2 Environmental Map of the Project Area

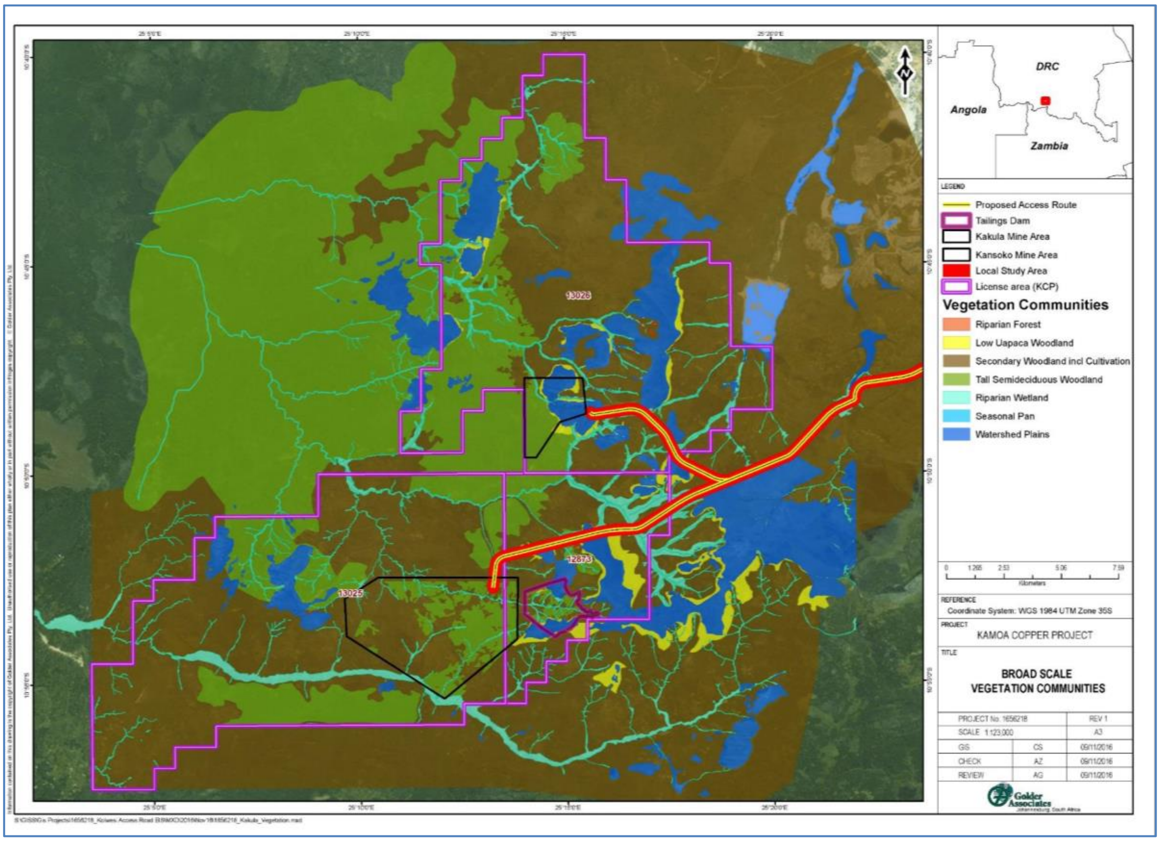


Figure by Golder, 2017.

The EIS update commenced in June 2012 with the collection of environmental, social and health data, stakeholder consultation and the development of a detailed scoping report and Terms of reference (ToR). The baseline data collection, scoping and ToR were completed in March 2014 (Golder, 2014). The work was put on hold in April 2014, pending finalisation of the project design for Phase 1, although Kamoa continued with baseline data collection. The EIS recommenced in the third quarter of 2016 with updates to the ToR, stakeholder consultation, the completion of the impact assessment and Environment and Social Management Plan (ESMP), further consultation and finalisation of the EIS update. The EIS update was submitted to government in January 2017 and approved in March 2017. The Project approved covers the two mining operations, the Kansoko Mine and the Kakula Mine. The mine development plan approved is for the production of 750,000 tonnes of copper concentrate per annum through the mining of a combined total of 8 million tonnes of copper sulphide ore. Government authorisation covers all the infrastructures of the two Kansoko and Kakula mines located within the three Exploitation Permit Nos. 12873, 13025, and 13026. It is noted that Kamoa commenced with initial development of a boxcut and decline in June 2014, based on the approved project description as presented in the original EIS (African Mining Consultants, 2011). The decline development was included in the EIS update. Furthermore, the Kakula boxcut and decline development was included in the approved March 2017 EIS update.

Third party developments in support of the Project include the development of the road from Kamoa-Kakula to Kolwezi airport (approved under the EIS update, 2017), and a power line from the national grid to the plant and various upgrades to existing hydroelectric schemes (approved under separate authorisation processes). In March 2014, a financing agreement was signed between Ivanhoe and the DRC's national electricity company, La Société Nationale d'Electricité (SNEL).

An Environmental Impact Study (EcoEnergie, 2013) was drafted for the power line; SNEL intends to update this EIS following project finalisation for submission to Government (an additional 12 km of the powerline servitude is required to be permitted). Regulatory environmental approvals are not required for the upgrades to the existing hydroelectric schemes.

As per the requirements of the DRC Mining Code – Kamoa generates the following additional environmental reports which are submitted to the regulator:

- Annual environmental reports.
- Bi-annual environmental third-party audits by a DRC certified environmental consulting company.
- Annual DPEM audits.

Internally the following reporting is undertaken:

- Weekly and Monthly reports; this presents a list of community incidents, grievances, stakeholder engagement, environmental incidents, environmental non-compliances, sanctions and fines and HSEC incidents.
- Weekly and Monthly monitoring reports covering – surface and ground water, dust fallout and noise.
- The KPS waste rock will be contained in a waste rock dump designed to handle the potentially acid generating pyritic siltstone.

20.1.3 Environmental Issues

Possible environmental issues that could materially affect the ability to extract the Mineral Resources or mineral reserves relating to current development/operations were determined utilising the following methodology:

- Review of Environmental and Social Reports.
- High level risk assessment of material issues utilising the following methodology:
  - Identification and listing of issues that could have an impact on Mineral Resource/Reserve extraction. These included permitting, legal non-compliance, highly sensitive environmental/social features and spatial/geographical features.
- Categorisation as follows:
  - None – issue will not impact mineral extraction.
  - Low – issue is unlikely to affect mineral extraction, would only result in disruption or delay for a short (less than one week) period of time and can easily be mitigated.
  - Medium – issue is likely to affect mineral extraction, would result in a moderate (one week to one month period) disruption or delay and can be mitigated.
  - High – issue is highly likely to affect mineral extraction, would result in extensive (>1 Month) disruption or delay to mineral extraction and cannot easily be mitigated.

From the application of the above approach, one low rated risk was identified for the current development/operation as per the initial phase of the Project:

- A number of options have been considered for the decline development to ensure that the workings remain dry, including the construction of dewatering boreholes, sealing thedecline with grouting and others. The abstracted water will be discharged into the receiving environment. No environmental issues are anticipated since the discharged water reflects unaffected water quality that is well within international and DRC effluent quality guidelines. The possible surface erosion due to the discharge will be mitigated by means of dedicated energy dissipation measures.

Two low risks were identified for future developments /operations (Phase 1):

- The EIS would need to be updated and approved prior to the development of major mining related infrastructure not included in the approved mine plan. These include the TSF, process plant, workshop and stores, contractor's camp and other associated infrastructure. If the EIS is not completed and approved on time, this could delay project development.
- Resettlement and compensation of persons who may be affected by physical and economic displacement. This needs to be completed prior to infrastructure development. If the Resettlement Action Plan (RAP) is not implemented prior to construction this could delay the Project. Further resettlement requirements will be identified at the completion of the EIA modelling to determine resettlement based on impacts.

The EIS update and RAP would need to be approved and the RAP implemented prior to Project construction. Kamoa Copper SA intends to commence with the Kakula RAP studies in May 2018 and this will specifically target the tailing facility storage infrastructure.

20.2 Waste, Tailings, Monitoring and Water Management

20.2.1 Waste

Kamoa has prepared the Terms of References to commission a Waste Management Plan. This work will be integrated as part of the EIS update. The overall objectives of the Waste Management Plan are:

- To manage waste in a manner that reduces, reuses, recycles and/or recovers the majority of waste with the aim of reducing waste to landfill. A firm has been identified that can recycle used oil, discussions are underway to examine partnership options.
- To identify options available for project waste management considering the remote location of, and limited access to, power for the Project.
- To provide a cost-benefit analysis of options for waste processing activities.
- To provide a detailed integrated plan to implement waste management prior to onset of construction activities.
- To identify innovative means of waste management at Kamoa which could include one or more community managed Small Medium Enterprise (SME) project(s).
- To ensure that waste management at Kamoa is conducted in a legally compliant manner.

20.2.2 Tailings Management and Disposal

In 2014 Epoch undertook a site selection study which determined that the site termed Mupenda is the preferred site for storing the tailings from Kansoko. The Mupenda site was selected over the other options for the following reasons:

- The topography and soil properties are such that it will not require expensive measures to both contain the tailings and prevent ground water and surface water contamination.
- It is able to cater for additional storage capacity should the LOM be extended or the tailings production rate increased.
- The risks associated with this site were deemed the lowest out of the other options due to the contaminated catchment downstream of the TSF due to the historical failure of the Potopoto TSF, as well as little to no people residing downstream of the site.
- High level costs showed that this site would have the lowest costs to construct a TSF.
- The TSF will be constructed as a double valley impoundment dam with a compacted earth impoundment wall. This will have the following features:
  - The TSF impoundment walls will be constructed as a downstream facility.

  - The wall is to be raised in 7 phases, where Phase 1 is at elevation 1465 m amsl and the last phase is at elevation 1495 m amsl.
  - A final phase (Phase 8), comprising a smaller upstream impoundment wall will be constructed on top of Phase 7 when the rate of raise reduces to <1.0 m/year.
  - The TSF has a total footprint area of 540 Ha, a maximum height of 52 m and a final rate of rise of <1.0 m/year.
- A Return Water Dam with a storage capacity of approximately 45,000 m^3.
- A concrete lined Return Water Sump with a water storage capacity of 2,000 m^3.
- A slurry spigot pipeline along the crest of the TSF.

The assumptions adopted for the Kansoko TSF are the following:

- Sufficient and suitable construction materials for the preparatory earthworks associated with the TSF can be sourced from the TSF basin and nearby borrow pits.
- The legislation that has been adopted for the purpose of this study is "Appropriate Best Practice Measures". "Appropriate Best Practice Measures", in this case, implies the use of the South African Tailings Disposal Facility Design Standards and Codes (i.e. SANS 0286:1998 – "Code of Practice for Mine Tailings") amongst others.
- Furthermore, the local DRC laws regulating TSFs have been taken into account, that stipulate:
  - Appropriate measures must be taken to ensure that no toxins from any tailings storage areas enter into the groundwater. Different requirements are applicable depending on the geochemical nature and toxicity of the tailings product.
  - Surface erosion problems shall be controlled by preferably planting vegetation. Erosion problems in unconsolidated materials shall be eliminated by reducing the hydraulic gradient. If materials of different particle grading's are placed in contact with each other, appropriate filter criteria must be observed.
  - It is understood that the tailings has been classified as a "leachable mine waste" by Golder Associates (Pty) Ltd (Golder), therefore areas where the in-situ material have a permeability greater than 1x10-6cm/s must be lined with an appropriate liner system, according to the DRC Regulations. A geochemical evaluation of a tailings sample generated from a pilot plant using Kamoa ore obtained from the geological exploration boreholes was conducted (Golder, 2015). The results from the Toxicity Characteristic Leach Procedure (TCLP) showed exceedances of copper and iron in relation to Annexure XI of the DRC Mining Regulations. This was consistent with the 20-week kinetic testing results. The tailings are therefore classified as a Leachable Mine Waste that require an engineered barrier if the underlying soil does not display a permeability of ≤1x10-6 cm/s over a depth of 3 m. The majority of the TSF footprint meets this criterion, apart from the Kalahari sand which has a permeability of 1x10-5 cm/s.
  - The area underlain by Kalahari Sand is approximately 55 ha along the north-east trending Chamilundu drainage line. A suitable engineered barrier to reduce the permeability over this area to meet the criterion will have to be constructed. Provisionally, an HDPE liner has been proposed. This will be reviewed with the next phase of the Project.

20.2.3 Environment Resources

The Kamoa Copper SA environmental management team comprises eight permanently employed staff members working within the five environmental management pillars shown in Table 20.1. The team is supported by external consulting expertise as required.

Table 20.1 Environmental Management Pillars – Kamoa

Biodiversity	Surveillance (Monitoring)	Waste Management	Performance and Compliance	EIS/RAP
Reclamation, rehabilitation and restoration	Water	4 R's (Reduce, Reuse, Recycle, Recover)	Land disturbance permit	Indicators
Characterisation and biodiversity	Noise	Solid Waste	Inspections	Monitoring and evaluation
Reforestation	Air Quality	Hazardous Waste	Incidents	Impact Management
Nursery	Vibration	Liquid Waste	External Audits	Exclusion Zone
Biodiversity Action Plan	Meteorology	Mine Waste	Other – oils, petrol, diesel management etc	Livelihood
–	–	Composting	–	Risk Management

20.2.4 Site Monitoring

Environmental, social and health baseline data collection and ongoing monitoring has been carried out within the study area since 2010. In summary, this includes the following:

- Climate – Kamoa currently collates meteorological data from Kolwezi Airport located approximately 25 km east of the Project and since 2010 has been recording meteorological data at a dedicated meteorological station on site and four dedicated rain gauges. The site meteorological station will be upgraded to a fully established professional station during 2016.
- Air Quality - Kamoa undertook an ambient air quality monitoring campaign at 24 sites from April to December 2012. The pollutant parameters monitored included total suspended particulates (TSP), nitrogen dioxide (NO_2), sulphur dioxide (SO_2) and ozone (O_3). Subsequent to the initial monitoring campaigns, Kamoa Copper SA is currently undertaking dust fallout monitoring at 10 monitoring sites. The dust fallout monitoring was initiated on 06 July 2013 and was undertaken on a monthly basis until the end of 2014. Results indicate that the air quality is very good. The site currently undertakes dust fallout monitoring of 9 sites 4 times a year.
- Noise and Vibration - A baseline noise monitoring campaign was carried out by African Mining Consultants (AMC) on seven occasions between November 2010 and September 2012. Ongoing noise monitoring is undertaken by Kamoa on a weekly basis. Results indicate noise levels are below guideline limits, except near villages which are caused by human activity not related to the project. Vibration monitoring was conducted in the box-cut area and Israel village during the blasting campaigns in 2015.

- Soils, Land Use and Land Capability – three soil, land use and land capability surveys have been undertaken for the Project since 2010 covering all infrastructure locations.
- Surface Water Hydrology – a total of 14 surface water sites are currently being monitored on a quarterly basis for both quality and flow. Initially from 2010 monitoring was on a monthly basis; this was changed to quarterly once seasonal variations were understood. Results indicate good water quality conditions apart from the Luilu River downstream of the failed Potopoto Tailings Dam which has been impacted by historical mining operations.
- Groundwater – A groundwater monitoring programme has been in place since 2010. The monitoring network was expanded to incorporate many of the boreholes drilled during the 2012 and 2013 PFS drilling programme. Monthly water level monitoring is undertaken at 64 boreholes located in the Project Area. Water level loggers are installed in 7 boreholes. Sampling for water quality monitoring is undertaken quarterly at 51 boreholes. Results indicate good water quality conditions, although some areas indicate pH levels lower than the recommended WHO standards due to the natural geological formation of the area.
- Geochemistry – testwork (leach tests and kinetic tests) is being undertaken to determine Acid Rock Drainage (ARD) and Metal Leaching risk of potential waste rock, tailings and run-of-mine (ROM) ore stockpiles as per the Global ARD Guidelines with results compared to DRC regulations to determine the required mitigation measures for the waste rock dump, TSF and ore stockpiles. Results will be presented in the updated EIS.
- Radiation – A once off radiation survey was undertaken in August 2012 and included a gamma survey Soil, water, sediment and vegetation sampling and airborne dust activity sampling taken at the proposed mining area. Results indicated normal radiation conditions and limited radiological risk.
- Ecological – aquatic, terrestrial, wetland and ecosystem goods and services evaluations have been undertaken over two seasons in 2012 and over one season in 2011 and in 2016. Ecological monitoring is undertaken on an ongoing basis. Two members of the environmental department are responsible for all issues regarding biodiversity. In 2014, Kamoa put in place a nursery aimed at future rehabilitation and restoration. Progressive rehabilitation has been adopted as a practice to ensure impact minimisation and understand best practices. A plot near the Kamoa camp has been reserved for reforestation purposes and an agreement is in process with nearby communities to replicate this initiative at the community level.
- Social – Three socio economic surveys have been undertaken in the Project area: by Kamoa in 2010 and 2011 and by Golder in 2013.
- Resettlement and Compensation – Land requirements for Phase 1 will result in the economic and/or physical displacement of approximately 80 to 100 households in two villages. This is for the construction of the Mupenda TFS. Additional resettlement needs will be determined during the EIA modelling and will include impacts based on noise, dust, access, and safety.
- Economics – macro- economic data (GDP, tax, income rates and employment levels) from secondary sources was collected for the DRC and Katanga province in 2013 by Golder for EIS purposes.

- Health – specific health information was collected for the Project in 2013 by Golder for EIS purposes.
- Archaeology – archaeological and cultural heritage surveys of proposed infrastructure development areas were undertaken in 2011 for the initial EIS and 2013 for the update of the EIS by Golder.

20.2.5 Water Management

The water demand for Kamoa Copper is estimated to be 9.1 ML/d for the 6 Mtpa mine and 18.2 ML/day for the 12 Mtpa production option. Groundwater from the lower basal sandstone regional aquifer is the preferred source of bulk water supply and will be obtained by groundwater from the Southern wellfield located between 6 and 8 km to the south-west of the mine site on the southern portion of the Makalu Dome, (Golder 2015). The bulk water supply will be augmented by the dewatering boreholes to be to be drilled along the line of the decline, and could be augmented by water from the Haute Luilu Dam (Golder, 2014); this latter is considered only as a long-term contingency. Potable water for the Project will also be obtained from the wellfield.

The numerical flow modelling undertaken as part of the hydrogeological study has indicated that groundwater ingress will be relatively limited, at an average inflow of approximately 7 l/s per km^2 of mining void, (Golder, 2014). In future the bulk water supply could be augmented by excess underground mining water make as the mine void increases in spatial extent.

Studies are currently underway to determine impacts on water sources and management plans to address these will be developed as part of the updated EIS. These will include the preparation of a stormwater management plan for the entire mining complex, sized to convey the 100-year flood peak, and the development of a mine water balance. The water balance will be used to size pollution control dams to meet the one in 100-year spill frequency.

20.3 Project Permitting

As per the applicable DRC mining law and regulations, mining licences are mandatory before carrying out any mine activity. Kamoa was issued with the following by the DRC competent authorities for Exploitation Permits 12873, 13025, and 13026:

- On 31 January 2012, the Department for the Protection of the Mining Environmental approved the EIS (African Mining Consultants, 2011).
- The Ministry of Mines issued three Exploitation Permits to Kamoa on 21 August 2012.
- The updated EIS (2017) was approved on 3 March 2017.

20.3.1 Financial Guarantee

In terms of the financial guarantee required by DRC Law, the EIS included an estimate of the total closure costs, amounting to US$8.1 M. The DRC Mining code requires payments as financial guarantees. Kamoa Copper SA has made the payments required to the end of 2017 totalling $495,000 as per the payment schedule. The closure cost estimate and financial guarantee provision will be updated during the course of the EIS update process.

20.4 Social and Community Related Requirements and Plans

Kamoa recognises the importance of effectively doing its business by incorporating within its day-to-day management all the stakeholders concerns. These are not only its obligations to mitigate or compensate local communities for environmental and social impacts caused by the project but a community investment as added-value investment built on local identified initiatives. Nevertheless, the two are interrelated components of a holistic approach for managing company-community relationships.

Kamoa has developed a Strategic Sustainable Development Plan SSDP whose overall goal is to form the basis of and guide implementation of social and economic measures during mine construction, operation and closure that will:

- Minimise negative environmental, social and economic impacts, and maximise benefits.
- Interlink environmental, social and economic dimensions to capitalise on opportunities and benefits.
- Leave a positive legacy beyond mine closure, and thus a contribution to sustainable development.

The timeframe of the SSDP is 5 years (2016 to 2020). The scope of the current SSDP is limited to:

- Primary affected people, i.e. those living within the mining concession.
- Secondary affected people, i.e. those living in the Musokantanda and Mwilu Chiefdoms but outside the mining concession.Regional, including Mutshatsha territory and Kolwezi town.

The Specific objectives of the SSDP are to:

- Invest time, expertise and resources to provide economic opportunity, improve the quality of life and foster goodwill in the communities living around Kamoa through locally relevant initiatives.
- Engage with relevant stakeholders including local authorities, communities and their representatives, inter-governmental and non-governmental organisations and other interested parties to support projects that benefit the communities associated with Kamoa's operations.
- Assist in creating sustainable cooperatives for the benefit of local people through partnerships thus contributing to addressing food security issues and assists in the building of self-sustaining economies in the communities and improving people's standard of living.
- Assist in creating strong and reliable Small and Medium Sized Enterprises (SMEs) for local and provincial economic development through a defined enterprise support system. Develop SMEs to act as potential local suppliers of goods and services to the mining sector.
- Develop robust Livelihoods/agriculture training facility that will contribute to the skills development, productivity and economic growth.

The existing Kamoa SDP has defined its investment into four main areas:

- Stakeholder engagement.
- Livelihoods creation and improvement through the Kamoa Sustainable Livelihoods Project.
- Community economic development.
- Community skill transfer.

A community-needs assessment was undertaken by Kamoa in 2014 within communities that will potentially be affected by its operations. The needs assessment provided indications on the most important concerns for these communities and their priorities. Community projects were then proposed based on the results of the need assessment exercise.

In 2015, the Lufupa and the Luilu sectors where the Kamoa Copper SA is carrying out its operations developed their own Local Development Plan with the assistance of local NGO (SADRI and Alternative Plus) financed respectively by Cordaid and GIZ.

From the recommendations made by Investissement Durable au Katanga (IDAK), Kamoa engaged the Lufupa sector and set up a workshop to align its SDP to the Lufupa LDP. The workshop took place from 10 to12 March 2016 at Musokantanda and involved community leaders, Government representatives, two mining companies (Kamoa Copper SA and Kalongo Mining) and two NGOs (SADRI and ACIDH). Based on outcomes of this workshop the SSDP has been updated to align with the key objectives and strategies of the Lufupa LDP (Table 20.2). Further updates will be undertaken once the Lufira sector LDP has been fully developed. A total of US$1,519,000 has been budgeted for the implementation of the SSDP.

Table 20.2 Social Management Pillars – Kamoa

Community Development	Economic and Livelihoods Development	Health, Safety and Human Rights	Stakeholder Engagement	ESHIA/RAP/SEMP	Regional Development
Education	Agriculture	HIV	Grievance management	Indicators	Partnerships
Literacy	Small and medium sized enterprises	Malaria	Donations	Monitoring and evaluation	Technical training
Rural Infrastructure	Micro-finance	Tuberculosis	Communication	Resettlement working group	Capacity building
Water	Technical and professional training	Nutrition	Cultural Heritage	Impact management	Health
Sport	Local employment procedure	Road Safety	Stakeholder mapping	Exclusion zone	Education
Community agreements	–	Support health services	Partnerships	Livelihood restoration	FIO implementation
Capacity Building	–	Human Rights awareness	Corporate social partnership	Risk Management	–
Electricity	–	–	–	–	–

20.4.1 Social / Community Issues

Adopting the risk assessment approach outlined in Section 20.1, no social or community issues were determined to have a moderate or high risk of material impact on the ability to extract the Reserves or Resources. It should be noted that local and national electoral issues in DRC might lead to political unrest which could impact mineral extraction.

20.4.2 Risks Identified by Kamoa

Through its ongoing risk assessment and evaluation as part of its Sustainability Management System (see Section 20.5), Kamoa has identified the following key risks and management strategies (Table 20.3).

Table 20.3 Kamoa Risk Assessment, July 2015

Risk Description	Consequence	Management Strategy
High level of expectations from the population.	Frustration and unsustainable dependency.	Public disclosure during the EIS update and implementation of the Stakeholder Engagement Plan (SEP). Communication consistencies. Kamoa Copper SA policies. Local Development Plans.
Employees: Strikes, sabotage.	Reputation, relationship with government deteriorates, financial loss, project delays.	Human Resources strategy. Local hiring procedure. Union relationship, talks between management and employees.
Permitting.	Project delays and financial impacts. Relationship with Government deteriorates.	Ongoing engagement.
Deterioration of the water, air and soil quality and deforestation.	Impacts on water, soil flora and fauna. Loss of social licence. Reputational issues.	Monitoring. Erosion control. Updating of the EIS. Implementation of the Environmental and Social Management Plan (ESMP).
Employment expectations.	Community blockades and loss of social licence.	Public disclosure during the EIS update and implementation of the Stakeholder Engagement Plan (SEP). Communication consistencies. Local hiring procedure.
Influx.	Poverty, pressure on natural resources, and pressure on existing community services could result in reputational issues and loss of social licence.	Updating of the EIS. Implementation of the ESMP. Housing strategy for workers. Demography tracking, grouping of social infrastructures that incite settlement.
Resettlement: Time and inadequate resettlement due to time constraints, previous survey has resulted in expectations.	Project delays and risks associated with reputation. Lack of compliance with IFC standards.	Completion of the EIS update and RAP in good time. Approval by board. Stakeholder Engagement Plan for RAP.
Pressure from local authorities and limited capacity.	Project delays, frustration, community mobilisation, unsustainable dependency, reputational issues.	Harmonise and align the Strategic Sustainable Development Plan to the Local Development Plans.

20.5 Sustainability Management System

In 2014, Kamoa put in place a Sustainability Management System comprising of the following areas:

- Sustainability Management System - Management System procedure.
- Policy Leadership and Commitment - Sustainability policy, Environmental and Social responsibilities and accountabilities.
- Hazard Identification and Risk Management - Risk Assessments and Risk Register.
- Legal and Other Requirements - Register of legal obligations. Sustainability objectives and targets, Social and Environmental Improvement Plans.
- Objectives Targets and Performance Management – specific indicators, reporting parameters.
- Training Awareness and Competence - Induction, training and awareness material, Training Needs Analysis, training attendance registers and records.
- Communication, Consultation and Participation - Sustainability team meetings records, Stakeholders Meetings register, Stakeholder Communications and Stakeholder Engagement Plan.
- Documentation and Document Control - templates to develop documents, document control process and register, records of Approval Request Forms.
- Operational Control - Relevant documentation to manage social and environmental aspects (e.g. waste management procedure, compensation rates, H&S plan for the communities, etc.).
- Change Management.
- Emergency Preparedness and Response - Records of emergency response exercises, link to the site Emergency Response Plan.
- Contractor Management - Specifications for contractors. Incidents and grievances process and supporting templates. Incidents and grievances.
- Incident and Grievances Reporting and Management registers - Records of incidents and grievances management (e.g. investigation reports and filled grievances forms).
- Monitoring Audits and Review – Monitoring programs and outcomes.

20.6 Mine Closure

The original EIS (African Mining Consultants, 2011) presented an initial framework closure plan. This work also included the determination of the mine closure costs that were based on market knowledge, past costing and the consultant's experience. This was then updated in 2017 for the latest EIS update.

The Kamoa 2017 PFS currently defines a 26-year mining plan; however, the resource is sufficiently large to support multiple expansion phases that could extend the life of the mine well beyond 26 years. The mine will undergo decommissioning and closure in accordance with DRC regulatory requirements at the time it is decided to close the mine.

Mine decommissioning and closure will be conducted with the following in mind:

- Creation/reinstatement of physical stable and lasting landforms.
- Protection of public health and safety.
- Limiting, and preferably obviating, predictable environmental effects, both physically and chemically.
- Reinstatement of meaningful next land use.
- Sustainability of the social programmes, including livelihoods and resettlement.
- Stakeholder engagement for closure.
- Reinstatement of meaningful land functionality.
- Optimisation of the possible social and economic benefits that could be derived from the mine in its closed state. If it is practicable, the mine will cede mine buildings, infrastructure, equipment and materials to the nearby communities to sustain/enhance local social and economic activity. This could also include the possible ongoing use of access roads created for the purpose of mining.

The key mining related infrastructure and related aspects that will require attention at mine decommissioning and closure include the following:

- Underground mine workings and related infrastructure.
- Waste Rock Dumps (WRD) and overburden spoil heaps.
- ROM pad and ROM stockpiles.
- Metallurgical Processing Facility.
- Workshops, stores and administration buildings.
- Tailings Storage Facility (TSF).
- Transport infrastructure such as site access roads, bridges and road drainage channels.
- Waste storage dams and mine site drainage systems/networks.

The decommissioning and closure of the above would in most cases follow routine practices such as removal of remaining contaminated soils and deep burying of these within the TSF before final rehabilitation, shaping and covering of outer slopes and upper surfaces of the WRD and remaining overburden piles, etc.

As underground mining methods will be followed, surface subsidence is possible. If surface subsidence occurs it should be limited and could be rectified by means of routine surface infilling, shaping and levelling.

The performance and success of the implemented closure measures will be checked and tracked by means of dedicated post closure inspection and monitoring programmes. The monitoring programmes will specifically focus on possible adverse effects on watercourses and groundwater within the zone of influence of the closed mine, reinstatement of landscape functionally (including vegetation establishment) as well as those aspects that pose potential adverse health risks and/or dangers to the public. The latter would include possible surface subsidence due to caving.

The above performance and success inspections and monitoring will be conducted by reputable independent third-party contractors. The outcomes of this work will be reflected in annual post-closure performance reports. These reports will be submitted to DPEM and made available to stakeholders as required. In those cases where the closure measures are not performing as designed, corrective action will be conducted.

The mine closure costs cover mine site decommissioning and closure measures as well as post closure inspections and monitoring as outlined above. The estimated full decommissioning and closure costs as at 2011 for the Project amount to US$8,122,375. This includes US$1,624,475 for closure management by independent third-party contractors (25% fee). The costs assume that rehabilitation and closure work is also carried out by third party contractors and that no revenue would accrue from the sale of mine equipment and/or demolition material to offset these costs.

It is noted that the current developments /operations only include the construction of a box-cut for the decline to the planned underground workings. A network of dewatering boreholes may be established to dewater the box-cut for construction to proceed. The estimated closure costs for this initial work equates to about 5% of the above estimated overall costs.

As part of the EIS update the decommissioning and closure plan and associated costs will be reviewed and updated to align with current generally accepted good practice and international standards in this regard.

21 CAPITAL AND OPERATING COSTS

21.1 Summary

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

Capital and operating costs have been estimated for each of the following areas:

- Additional drilling.
- Underground mining.
- Additional power.
- Temporary facilities.
- Infrastructure.
- Concentrator.
- Indirect Costs.
- General and Administration.
- Rail.
- Transport.
- Closure.

Table 21.1 summarises unit operating costs, whilst Table 21.2 provides a breakdown of operating costs on a per tonne basis.

Table 21.1 Unit Operating Costs

	US$/lb Payable Cu		
	Years 1–5	**Years 1–10**	**LOM Average**
Mine Site	0.62	0.57	0.64
Transport	0.47	0.47	0.47
Treatment and Refining Charges	0.18	0.19	0.19
Royalties and Export Tax	0.22	0.22	0.22
Total Cash Costs Before Credits	1.49	1.44	1.51

Table 21.2 Operating Costs

	Total LOM	Years 1–5	Years 1–10	LOM Average
	US$M	US$/t Milled		
Site Operating Costs				
UG Mining	3,490	31.42	28.47	27.88
Processing	1,469	11.50	11.72	11.74
Tailings	29	0.30	0.23	0.23
General and Administration	774	7.33	6.04	6.18
SNEL Discount	-191	-2.18	-2.19	-1.48
Customs	82	0.73	0.67	0.66
Total	**5,654**	**49.11**	**44.95**	**45.21**

The capital costs for the project are summarised in Table 21.3.

Table 21.3 Capital Cost Summary

Capital Costs (US$M)	Initial Capital US$M	Expansion Capital US$M	Sustaining Capital US$M	Total US$M
Mining				
Underground Mining	311	–	806	1,117
Capitalised Pre-Production	4	–	–	4
Subtotal	315	–	806	1,121
Power and Smelter				
Power Supply Off Site	71	–	–	71
Capitalised Power Cost	1	–	–	1
Subtotal	**72**	**–**	**–**	**72**
Concentrator and Tailings				
Plant Capex	146	84	172	402
Tailings	21	95	–	116
Subtotal	**167**	**179**	**172**	**518**
Infrastructure				
General Infrastructure Capex	110	–	83	193
Other Infrastructure	35	–	26	61
Rail	–	48	–	48
Subtotal	**145**	**48**	**109**	**302**
Indirects				
EPCM	70	34	21	125
Owners Cost	79	20	8	107
Closure	–	–	76	76
Subtotal	149	54	106	308
Capital Expenditure Before Contingency	848	280	1,193	2,321
Contingency	156	68	141	365
Capital Expenditure After Contingency	1,004	348	1,334	2,686

21.2 Underground Mining Cost Estimates

This section describes the parameters, exclusions and the capital and operating cost basis of estimates to support the Kamoa 2017 PFS 26-year mine plan. Unit costs are based on the most recent cost information from similar projects and adjusted where required to fit the mine plan. All costs are based on 2017 US$.

21.2.1 Underground Capital Costs

The total capital cost includes both pre-production and sustaining capital. Pre-production capital includes all direct and indirect mine development and construction costs prior to the start of feed through the processing plant. The cost of initial mining equipment purchased by Ivanhoe for use by the Contractor for the pre-production development is also included. After the initial development is completed by the underground Contractors, the equipment fleet used for pre-production will be used for sustaining mine development activities.

Sustaining capital is comprised of ongoing capital development and construction as well as mobile equipment rebuild and replacement costs. A summary of capital costs can be found in Table 21.4.

Table 21.4 Summary of Underground Pre-production and Sustaining Capital Costs

Description	Pre-production (US$M)	Sustaining (US$M)	Total (US$M)
Contractor Costs			
Box-cut and Portal	0.9	0.0	0.9
Main Declines	10.6	0.0	10.6
Sud Declines	12.9	1.1	14.0
Centrale Declines	21.9	22.8	44.8
Ventilation Systems	22.2	20.5	42.7
Surface Infrastructure Facilities	4.7	0.0	4.7
Underground Infrastructure and Equipment	23.0	9.7	32.7
Mining	170.9	86.9	257.8
Vertical Ventilation Raises	3.2	14.6	17.7
Mobilization	6.7	0.9	7.6
Contractor Labour Uplift	16.2	1.7	17.9
Indirect Costs	60.2	7.8	68.0
Margins	46.9	6.1	52.9
Subtotal Contractors Costs	**400.3**	**172.1**	**572.4**
Owner Costs			
Fixed Equipment*	9.5	1.8	11.3
Fixed Equipment Spares	2.8	0.0	2.8
Mobile Equipment	129.0	319.6	448.6
Mobile Equipment – Initial Spare Parts	3.9	0.0	3.9
Owners Indirects	0.0	21.6	21.6
Engineering, Procurement, and Construction Management	30.0	6.4	36.4
Owner's Team	12.0	1.6	13.6
Power – Utility	6.1	0.0	6.1
Subtotal Owner Costs	**193.3**	**351.0**	**544.3**
Total Contractor and Owner Costs	**593.6**	**523.1**	**1116.7**
Contingency	105.8	72.7	178.5
Total Capital Costs	**699.4**	**595.7**	**1,295.1**

Fixed equipment for estimated items is included with construction costs.

The underground capital costs were estimated for the following:

- Portal.
- Underground Development – declines and primary development.
- Mobile Equipment – purchase, rebuild, and replacement.
- Fixed Equipment – including rock handling conveyors and tips.
- Surface Materials Handling Facilities with Boreholes (explosives, fuel and lube, concrete/shotcrete).
- Initial Electrical, Control, Communications, and Instrumentation Systems.
- Main Workshop with Offices and Stores.
- Underground Materials Handling Facilities (explosives, fuel and lube, concrete/shotcrete).
- Ore Bins with Feeders and Belts.
- Piping Services and Water Handling.
- Dewatering System.
- Ventilation Raises, Fans, Controls.
- Mine Air Refrigeration
- Mine Management Owners Team.
- Training of Underground Miners during the Pre-Production Period.
- Contingency Mining Cost.

21.2.2 Underground Operating Costs

Unit operating costs were prepared for room-and-pillar stoping and controlled convergence room-and-pillar stoping. Annual operating costs were generated based on the tonnes produced each year.

The underground operating costs were estimated for the following:

- Access Development for Controlled Convergence Room-and-Pillar.
- Production Direct Costs.
- Materials Handling Operation and Maintenance.
- Ground Support Rehabilitation.
- Dewatering.
- Ventilation and Refrigeration.
- Engineering / Mining Stores.
- Training.
- Indirect Operating Costs - not directly allocated to production.

- Power Costs.
- Undefined Allowance.

A summary of the 26-year mine plan total and average operating costs per tonne of ore is shown in Table 21.5.

Table 21.5 Underground Operating Cost Summary

Description	Unit Cost (US$/t of Total Ore)
Secondary Ore Development	1.73
Room-and-Pillar*	0.26
Controlled Convergence Room-and-Pillar*	11.13
Pillar Recovery*	4.36
Contractor Demobilization	0.06
Contractor Labour Uplift	0.08
Contractor Indirect Costs	0.39
Contractor Margins	0.31
Total Production Direct Operating Costs	**18.32**
Ventilation	0.07
Cooling Plant	0.01
Dewatering	0.14
Conveyors and Tips	1.01
Power	1.59
Indirect Operating Costs	5.24
Undefined Allowance	1.32
Total Production Indirect Operating Costs	**9.37**
Total Operating Cost	**27.69**

* Note: Costs weighted against total production tonnes.

21.3 Concentrator Costs

The capital and operational costs for the concentrator were prepared for the Kamoa 2017 PFS and are described below.

21.3.1 Concentrator Capital Cost Estimation Basis

Capital costs are defined as the expenditure required during the design, construction and commissioning phases of the project. This includes all costs associated with labour, construction, plant and equipment, bulk materials, other materials, permanent equipment, sub-contracts, packaging, transportation, loading, off-loading, strategic spares and capital indirect costs which contribute to the physical construction of the project.

21.3.1.1 Estimating - General

The following inputs and documents were identified and used in compiling the estimate:

- Process design basis.
- Site plot plans.
- Block flow diagrams.
- Process flow diagrams.
- Mechanical equipment list.
- Battery limits as described in the study documentation.

Costs have been estimated for the following disciplines:

- Earthworks.
- Civil works.
- Structural steel fabrication, supply and erection.
- Platework fabrication, supply and erection.
- Mechanical equipment supply.
- Mechanical equipment installation.
- Pipework fabrication, supply and erection.
- Electrical and C&I supply and erection.
- Transportation to site.
- EPCM services.
- First fills and spares.
- Infrastructure buildings.

21.3.1.2 Major Mechanical Equipment

Short-form enquiries were prepared and issued to three vendors for all major mechanical equipment. This category represented more than 90% of the total process plant mechanical equipment supply costs and included the following:

- Crushers.
- Feeders and screens.
- Conveyors.
- Ball mills and relining equipment.
- Regrind mills.
- Cyclones.
- Flotation Cells and Blowers.
- Slurry and Froth pumps.
- Thickeners.
- Concentrate filters.
- Bulk bagging system.

The installation costs for mechanical equipment were factorised from the supply costs and allowances were made for vendor installation, supervision and commissioning as appropriate.

21.3.1.3 Earthworks and Roads

Limited bulk earthworks have been allowed for as part of the civil bulk quantity estimate.

21.3.1.4 Surface Facilities and External Infrastructure

The following surface facilities are included in the cost estimate:

- Plant water services (including plant raw, potable, gland service and process water).
- Air services (including blower and compressed air).
- Plant pipe racks.

Furthermore, the following facilities are included in the overall estimate, within the plant infrastructure category:

- Tailings Storage Facility (TSF) pipeline (19,600 m for 2 lines).
- Return water pipeline (10,400 m).
- Borehole supply pipeline (10,450 m).
- In-plant roads.
- Plant vehicles.
- Sewerage treatment plant.
- Fencing.
- Infrastructure buildings.
- Substations.
- Camps.
- Dams.
- Temporary and backup power.
- Road/rail infrastructure.

21.3.1.5 Structural Steelwork

Structural steel material take-offs were developed from layout drawings. Rates for steelwork supply and fabrication were taken from database rates.

21.3.1.6 Civil Works

Civil bulk quantities were developed from layout drawings of the plant areas. Rates for civil works were obtained from MDM's database.

Preliminary and General (P&G) costs have been quantified as part of the civil summary.

21.3.1.7 Piping, and Valves

The process plant piping and valves cost estimates were factorised as a percentage of the mechanical supply cost.

21.3.1.8 Electrical, Control and Instrumentation

Budget quotations were obtained for major electrical equipment.

The pricing of all other electrical, control and instrumentation item costs were factorised from mechanical equipment costs.

The installation cost for the quoted major electrical equipment was obtained from vendors where vendor installation is required, or based on rates from similar projects in the MDM database.

21.3.1.9 Transportation

Load estimates and shipping and transport budget quotes for delivery to site were based on MDM in-house data.

21.3.1.10 EPCM

EPCM costs were built up from first principles, based on the project execution schedule and estimated based on MDM's current personnel rates for 2015.

21.3.1.11 Spares and Consumables

Costs were included in the estimate to cover operating, strategic and commissioning spares for the mechanical and electrical equipment. Allowances were made for first fills.

Concentrator Operating Costs

The operating cost estimate includes the fixed (labour and maintenance) costs and variable costs components (reagents, grinding media and power costs). The operating costs are expressed in United States Dollar (USD) per tonne milled. The operating cost figure excludes rehabilitation, mining, insurance costs, import duties and all other taxes.

The sources of information and assumptions are as follows:

- Vendor information and quotations.
- Plant labour rates and staffing levels as supplied by Kamoa Copper SA.
- Power cost ($0.0569/kWh) supplied by Kamoa Copper SA.
- MDM Technical Africa (Pty) Ltd (MDM) knowledge and experience.

21.3.1.12 Consumables

Crushing and Grinding

The consumables for the crushing and grinding sections include screen panels, crusher liners, mill liners and grinding media. The liner wear rate and steel ball consumption rate are estimated using the Bond abrasion index. The regrind mill ceramic media consumption is based on a vendor supplied rate which is referenced to the regrinding power consumed.

Flotation

The main flotation consumables are reagents and the consumption rates are based on the testwork performed by XPS. Reagent prices are provided by vendors.

Thickening

Flocculant consumption rates are assumed as no vendor settling testwork on concentrates or tailings has been conducted.

21.3.1.13 Utilities

Power

Power consumption is based on operating power estimates of the equipment in the MEL and using estimated operating time for that equipment.

Water

As water is supplied by bores or pumping from underground workings the cost of water is the cost of the power required to deliver it to the plant. These costs are in the power estimates.

21.3.1.14 Maintenance

A simple 5% factor has been applied to the overall mechanical equipment cost to provide a Kamoa 2017 PFS level maintenance cost estimate.

21.3.1.15 Transport for Consumables

Transport costs for delivering reagents and grinding media to site have been provided by the vendors.

21.3.1.16 Labour

The labour cost estimate is based on the labour rates and personnel numbers provided by Kamoa Copper SA.

The labour structure assumes a strong day shift presence in the plant when the bulk of the maintenance as well as all reagents off loading and make-up activities will be completed.

21.3.1.17 On-Site Laboratory

The on-site laboratory is to be operated by SGS under contract and, as such, the rates have been supplied by SGS.

21.4 Tailings Storage Facility

21.4.1 TSF Capital Cost

The capital costs associated with the TSF have been estimated by Epoch to an accuracy of +/-25% and have been based on contractor rates. Epoch has provided the following qualifications to their estimate:

- Preliminary and General costs (P&G's) accounting for 20% of the total works. Based on DRC experience, this value may be as high as 30%–40%.
- No allowance for escalation has been made.
- The above costs exclude provisions for:
  - Pumps.
  - Mechanical and electrical and instrumentation components.
  - Pump stations.
  - Slurry and return water pipelines between the TSF and the Plant.

Closure costs have been assumed to occur after production ceases. However, some closure costs may be incurred earlier if there are opportunities for progressive TSF rehabilitation.

21.4.2 TSF Operating Cost Estimate

The operating costs associated with the TSF have been estimated with allowances for the following:

- Tailings deposition and operations management.
- General works associated with the TSF.
- Consulting services.

21.4.2.1 Aftercare and Maintenance Requirements

On completion of the final rehabilitation and closure works, an aftercare and maintenance program will be enacted to ensure that the closure measures are robust, have performed adequately and that no further liabilities arise. The aftercare period is normally not less than 5 years but can extend into decades depending on the physical and chemical characteristics of the facility. The aftercare and maintenance program for is assumed to include:

- Periodic inspection of the cover and vegetation for signs of erosion damage and failures of the vegetation establishment process;
- Repairs and amendments to the closure works as necessary;
- Re-planting of areas of vegetation where required;
- Periodic inspection and monitoring to confirm the effectiveness of the closure works in achieving the stated closure objectives, including:
  - Collection and analysis of ground and surface water samples;
  - Measuring of phreatic surfaces within the TSF and assessment of the overall structural stability of the facility; and
  - Inspections of stormwater decant facilities for signs of damage.

No allowance has been made for the treatment of water that will need to be discharged into the environment from the TSF after closure as treatment is assumed to be unnecessary.

Aftercare costs were estimated for a period of two years following mine closure. This cost is indicative and has been based on closure cost estimates undertaken for similar operations, by Epoch.

21.5 Bulk Water Supply Capital and Operating Costs - Kamoa Wellfield

The following assumptions are made with regard to pumps, pipelines and associated infrastructure:

- The Mine will provide electrical power at each pump station.
- Limited civil work will be required to install the pumps at all the required places.
- Submersible pumps will be acceptable to be used.
- Stainless Steel pumps will only be required at the Pollution Control Dams (PCDs).
- Pipes can be laid on top of the ground.
- Joining of Pipes can be done with continuous welding.

21.5.1 Wellfield Development Capital and Operating Costs

A number of production boreholes are required to supply the estimated 9.1 ML/d for the Kamoa 2017 PFS production scenario.

Borefield capital is expensed ahead of production as start-up capital, while the operating, maintenance and energy costs will be incurred commencing in production Year 1.

21.5.2 Stormwater Management Plan

The following assumptions were made when developing the cost estimates for the stormwater management plan:

- South African construction rates were escalated 20%.
- Petrol and labour are included in the rates.
- No allowances for other escalations.
- P&G allowance of 15%.
- Engineering rates were assumed to be 12.5% of the total capital costs.
- The soil characteristics are assumed to be suitable for construction of the PCD walls and the berms alongside each channel.

Work not allowed for in the schedule of quantities and rates include:

- Box and key cut quantities and costs. The depths and configuration can only be finalised during the detailed and construction phases of the project.
- Hard excavation and blasting.
- All electrical, instrumentation and power supply items.
- All taxes (in country taxes, etc.).
- Costs for detailed design, tender documentation, code of practice, operation manual, quality assurance.
- Costs for any additional studies.

21.6 Owner's Cost

Ivanhoe have prepared a budget for Owners costs. The costs include costs for the following items:

- Office and General Expenses.
- Maintenance.
- Equipment and Sundry.
- Fuels and Utilities.
- Other Offices.
- Insurance and Insurance Taxes.
- IT Hardware and Software.
- Personnel Transport.
- Training.
- Communications.
- Licences and Land Fees.
- Labour Expatriate.
- Labour Congolese.
- Accommodation and Messing.
- Medical Support.
- Expatriate Flights.
- Light Vehicles.
- Environmental.
- Community Development.
- Banking and Audit Fees.
- Legal and Consultants.
- Studies.
- Resettlement.
- Capitalised General and Administration costs.

21.7 Power Infrastructure Rehabilitation and Upgrade

The costs of the power plants rehabilitation have been estimated by Stucky Ltd (Stucky) in its power study and updated by Kamoa Copper SA in 2015.

These estimated costs are based on equipment suited to the region.

Based on the June 2011 Memorandum of Understanding (MOU) with SNEL, the capital cost of the rehabilitation will be financed by Ivanhoe through a loan to SNEL. The loan including interest will be repaid by SNEL through a deduction from Ivanhoe's monthly power bills incurred over the life of the mine. For the financial analysis this has been assumed to be a 40% discount to the power charges and results in the discount being applied for 14 years from commencement of production.

21.8 Concentrate Transport Operating Costs

A phased logistics solution is proposed in the Kamoa 2017 PFS. Initially the corridor between southern DRC and Durban in South Africa is viewed as the most attractive and reliable export route. As soon as the railroad between Kolwezi and Dilolo, a town near the DRC Angolan border, is rehabilitated, Kansoko's production is expected to be transported by rail to the port of Lobito in Angola. The total costs including all fees and charges for the southern transport route were estimated to be US$356/t and for the western route via Lobito to be US$323/t.

21.9 Closure Costs

An allowance has been made for Closure costs in the financial model. This equates to 10% of all capital expenditure excluding Mining, Power and Indirect costs.

21.10 Comments on Section 21

For Underground Mining, costs were estimated at a prefeasibility study level of accuracy, with unit costs based on the most recent cost information from similar projects and adjusted where required to fit the mine plan.

Construction costs for the process plant have been developed based on using predominantly Chinese Contractors. The viability of utilising Chinese labour will be investigated during the next phase of this project.

Infrastructure and Plant estimate quantities were obtained from models and Bills of Quantities in 76% of the cases. Rates were obtained via budget quotes from vendors in 52% of the cases.

In the QPs' opinion the work completed adequately supports this level of study estimate.

22 ECONOMIC ANALYSIS

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

22.1 Financial Results Summary

The Reserve Case described in the study is for the construction and operation of a long-term underground mine, concentrator processing facilities, and associated infrastructure. The mining rate and concentrator feed capacity is 6 Mtpa. The production scenario schedules 125.2 Mt at 3.81% Cu over 26 years, producing 11.4 Mt of copper concentrate, containing 4,178 kt of copper in concentrate, during the life-of-mine. The economic analysis uses a long-term price assumption of US$3.00/lb for copper. The basis of the operational framework of the mine, used in the economic analysis, is DRC legislation and general industry terms. The economic analysis returns an after tax Net Present Value (NPV) at an 8% discount rate of US$2,063 M. It has an after tax internal rate of return (IRR) of 24.2% and a payback period of 5.0 years. The life-of-mine average total cash cost is US$1.51/lb of copper.

The key results of the Study are summarised in Table 22.1.

Table 22.1 Kamoa 2017 PFS Results Summary

Item	Unit	Total
Total Processed		
Quantity Milled	kt	125,182
Copper Feed Grade	%	3.81
Total Concentrate Produced		
Copper Concentrate Produced	kt (dry)	11,405
Copper Recovery	%	87.52
Copper Concentrate Grade	%	36.63
Contained Metal in Concentrate	Mlb	9,211
Contained Metal in Concentrate	kt	4,178
Peak Annual Contained Metal in Concentrate	kt	245
10 Year Average		
Copper Concentrate Produced	kt (dry)	487
Contained Metal in Concentrate	kt	178
Mine Site Cash Cost	US$/lb	0.57
Total Cash Cost	US$/lb	1.44
Key Financial Results		
Peak Funding	US$M	1,070
Initial Capital Cost	US$M	1,004
Expansion Capital Cost	US$M	348
Sustaining Capital Costs	US$M	1,334
LOM Average Mine Site Cash Cost	US$/lb Cu	0.64
LOM Average Total Cash Cost	US$/lb Cu	1.51
Site Operating Costs	US$/t Milled	45.21
After-Tax NPV8%	US$M	2,063
After-Tax IRR	%	24.2
Project Payback Period	Years	5.0
Initial Project Life	Years	26

Table 22.2 summarises the financial results, whilst Table 22.3 summarises mine production, processing, concentrate, and metal production statistics.

Table 22.2 Financial Results for Kamoa 2017 PFS 6 Mtpa

Net Present Value (US$M)	Discount Rate	Before Taxation	After Taxation
	Undiscounted	10,512	7,441
	4.0%	5,560	3,874
	6.0%	4,104	2,824
	8.0%	3,049	2,063
	10.0%	2,273	1,503
	12.0%	1,694	1,086
Internal Rate of Return		28.3%	24.2%
Project Payback Period (Years)		4.8	5.0

Table 22.3 Mine Production and Processing Statistics for Kamoa 2017 PFS 6 Mtpa

Item	Unit	Total LOM	Years 1–5	Years 1–10	LOM Average
Total Processed					
Quantity Milled	kt	125,182	3,533	4,777	4,815
Copper Feed Grade	%	3.81	4.24	4.20	3.81
Total Concentrate Produced					
Copper Concentrate Produced	kt (dry)	11,405	358	487	439
Copper Recovery	%	87.52	88.11	88.70	87.52
Copper Concentrate Grade	%	36.63	36.85	36.48	36.63
Contained Metal in Concentrate					
Copper	Mlb	9,211	291	392	354
Copper	kt	4,178	132	178	161
Payable Metal					
Copper	Mlb	8,884	280	378	342
Copper	kt	4,030	127	171	155

22.2 Democratic Republic of the Congo Fiscal Environment

A Mining Code (Law No. 007/2002 of 11 July 2002) (2002 Mining Code) governs prospecting, exploration, exploitation, processing, transportation, and the sales of mineral substances.

22.3 Model Assumptions

22.3.1 Pricing and Discount Rate Assumptions

The Project level valuation model begins on 1 December 2017. It is presented in 2017 constant dollars; cash flows are assumed to occur evenly during each year and a mid-year discounting approach is taken.

The copper price used for the evaluation is US$3.00/lb copper. This is considered to be reasonable based on industry forecasts and prices used in other studies. The product being sold is copper concentrate and payment terms for the copper assume that the life-of-mine average payable copper concentrate is 96.45%.

The copper concentrate attracts an $80 per tonne treatment charge and refining charge of US$0.08/lb copper. The copper concentrate transport charge (including provincial road taxes and duties but excluding the provincial concentrate export tax and DRC export tax) to the customer is assumed to be US$356/t via road (for transport via Durban) for the first two years of production and thereafter US$323/t via rail (via Lobito).

22.3.2 Taxation

The DRC Mining Code provides for all the taxes, charges, royalties, and other fees. Ivanhoe engaged KPMG South Africa, to report on which tax assumptions are applicable to an operating mine in the DRC. Only material taxes that would have an impact on the financial model have been considered and require confirmation.

In the analysis, carry balances such as tax and working capital calculations are based on nominal dollars and outputs are then deflated for use in the integrated cash flow calculation. The working capital assumptions for receivables, payables are 6 weeks and 6 weeks. These assumptions are preliminary and will need to be verified in later studies.

22.3.3 Royalties

A company holding a mining exploitation licence is subject to mining royalties. The royalty is due upon the sale of the product and is calculated at 2% of the price received of non-ferrous metals sold less the costs of transport, analysis concerning quality control of the commercial product for sale, insurance and marketing costs relating to the sale transaction.

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

22.3.4 Key Taxes

The DRC Mining Code provides for all the taxes, charges, royalties, and other fees. Ivanhoe engaged KPMG South Africa, to report on which tax assumptions are, applicable to an operating mine in the DRC. Only material taxes that would have an impact on the financial model have been considered and require confirmation. The key taxes identified by KPMG are listed below.

General Corporate Taxation

Companies that are the holders of mining rights are subject to tax at 30% on net income and withholding tax on distributions are subject to 10% tax at the shareholder's level. In addition, as from 01 January 2014, the minimum amount of tax payable by mining companies in a year is 1% of the calculated revenue for that specific year ("Minimum Tax Amount").

Funding / Thinning Capitalisation

No thin capitalisation rules apply in the DRC.

Tax Holidays

The DRC tax legislation does not currently provide for any tax holiday incentives.

Tax Losses

The aggregate exploration expenditure may be claimed.

Taxes on Products Sold

The tax rates will not change depending on whether concentrate or refined products are ultimately sold.

Depreciation

Specific mining assets dedicated to mining operations, with useful lives between 4 and 20 years are depreciated as follows:

- First year: 60% depreciated based on the cost of the asset.
- For subsequent years: a declining balance depreciation is applied based on the tax years remaining over the life of the mine.

Non-mining assets are depreciated in accordance with the common law. The common low provides different depreciation rates for various assets, e.g. 10 years for plant and equipment.

VAT

VAT came into effect in the DRC in January 2012. VAT is levied on all supplies of goods and services at a rate of 16% and is not levied on any capital asset movements.

Customs/Import Duties

Customs duty will be applied separately to capital (Pre-Production 2%, Post Production 5%) and operating costs (3%) for direct cost line.

Export Taxes

National Export Tax

The fee is limited to 1% of the value of the export.

Provincial Export Tax on Concentrate

A provincial tax on the export of concentrate is levied on a per tonne basis and equates to US$100/t concentrate exported.

Provincial Export Road and Infrastructures Renovation Tax

A provincial export tax levied on any product exported by road is also levied on a per tonne basis at a rate of US$50/t. Copper concentrate will be exported by road to neighbouring countries, and will thus be subject to the Road tax.

Withholding Taxes

A Withholding tax at the rate of 14% on services supplied by foreign companies established offshore to onshore companies applies. Mining companies are liable for movable property withholding tax at a rate of 10% in respect of dividends and other distributions paid. Non-mining companies are subject to withholding tax of 20%.

Dividend Distributions/Interest Repayments

Any dividend distributions made to Ivanhoe, as well as the DRC government will attract a withholding tax of 10%. A withholding tax of 20% applies if the loan is denominated in local DRC currency. If the loan is however denominated in foreign currency no withholding tax is payable. Interest payments to any local intermediate and holding companies attract a withholding tax of 20%.

Exceptional Tax on Expatriates

In the DRC, an employer is liable for the exceptional tax on expatriate's remuneration at a rate of 25%. Mining companies are subject to 10%. It is determined in terms of the salaries generated by the work carried out in the DRC, and is deductible for purposes of calculating the income tax payable.

22.4 Kamoa 2017 PFS Overview and Results

The Reserve Case described in the study is for the construction and operation of a long-term underground mine, concentrator processing facilities, and associated infrastructure. The mining rate and concentrator feed capacity is 6 Mtpa. The production scenario schedules 125.2 Mt at 3.81% Cu over 26 years, producing 11.4 Mt of copper concentrate, containing 4,178 kt of copper in concentrate, during the life-of-mine.

The economic analysis used a long-term price assumption of US$3.00/lb for copper. The basis of the operational framework of the mine, used in the economic analysis, is DRC legislation and general industry terms. The economic analysis returns an after tax Net Present Value (NPV) at an 8% discount rate of US$2,063 M. It has an after tax internal rate of return (IRR) of 24.2% and a payback period of 5 years. The life-of-mine average total cash cost is US$1.51/lb of copper.

The key results of the Study are summarised in Table 22.4. Table 22.5 summarises the cash flow analysis, whilst Table 22.6 summarises mine production, processing, concentrate, and metal production statistics.

Table 22.4 Kamoa 2017 PFS Results Summary

Item	Unit	Total
Total Processed		
Quantity Milled	kt	125,182
Copper Feed Grade	%	3.81
Total Concentrate Produced		
Copper Concentrate Produced	kt (dry)	11,405
Copper Recovery	%	87.52
Copper Concentrate Grade	%	36.63
Contained Metal in Concentrate	Mlb	9,211
Contained Metal in Concentrate	kt	4,178
Peak Annual Contained Metal in Concentrate	kt	245
10 Year Average		
Copper Concentrate Produced	kt (dry)	487
Contained Metal in Concentrate	kt	178
Mine Site Cash Cost	US$/lb	0.57
Total Cash Cost	US$/lb	1.44
Key Financial Results		
Peak Funding	US$M	1,070
Initial Capital Cost	US$M	1,004
Expansion Capital Cost	US$M	348
Sustaining Capital Costs	US$M	1,334
LOM Average Mine Site Cash Cost	US$/lb Cu	0.64
LOM Average Total Cash Cost	US$/lb Cu	1.51
Site Operating Cost	US$/t Milled	45.21
After-Tax NPV8%	US$M	2,063
After-Tax IRR	%	24.2
Project Payback Period	Years	5.0
Initial Project Life	Years	26

Table 22.5 Financial Results for Kamoa 2017 PFS 6 Mtpa

Net Present Value (US$M)	Discount Rate	Before Taxation	After Taxation
	Undiscounted	10,512	7,441
	4.0%	5,560	3,874
	6.0%	4,104	2,824
	8.0%	3,049	2,063
	10.0%	2,273	1,503
	12.0%	1,694	1,086
Internal Rate of Return		28.3%	24.2%
Project Payback Period (Years)		4.8	5.0

Table 22.6 Mine Production and Processing Statistics for Kamoa 2017 PFS 6 Mtpa

Item	Unit	Total LOM	Years 1-5	Years 1-10	LOM Average
Total Processed					
Quantity Milled	kt	125,182	3,533	4,777	4,815
Copper Feed Grade	%	3.81	4.24	4.20	3.81
Total Concentrate Produced					
Copper Concentrate Produced	kt (dry)	11,405	358	487	439
Copper Recovery	%	87.52	88.11	88.70	87.52
Copper Concentrate Grade	%	36.63	36.85	36.48	36.63
Contained Metal in Concentrate					
Copper	Mlb	9,211	291	392	354
Copper	kt	4,178	132	178	161
Payable Metal					
Copper	Mlb	8,884	280	378	342
Copper	kt	4,030	127	171	155

Figure 22.1 and Figure 22.2 depict the processing, concentrate and metal production, respectively.

Figure 22.1 Plant Feed Processing for Kamoa 2017 PFS 6 Mtpa

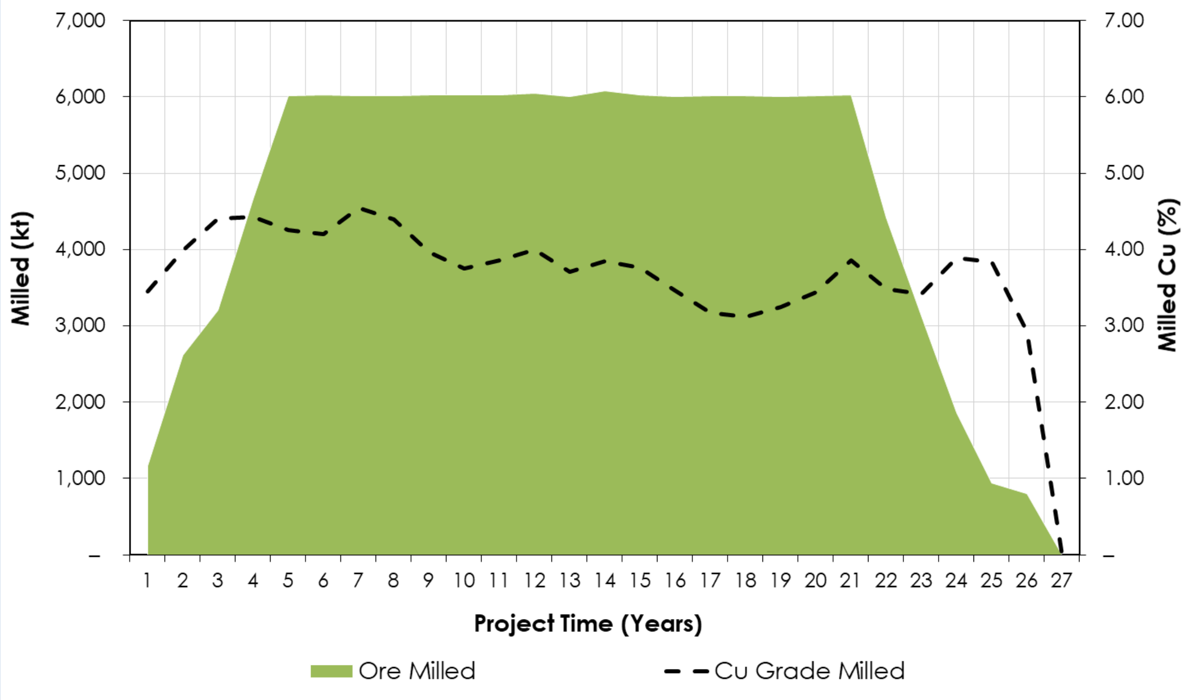


Figure by OreWin, 2016.

Figure 22.2 Concentrate and Metal Production for Kamoa 2017 PFS 6 Mtpa

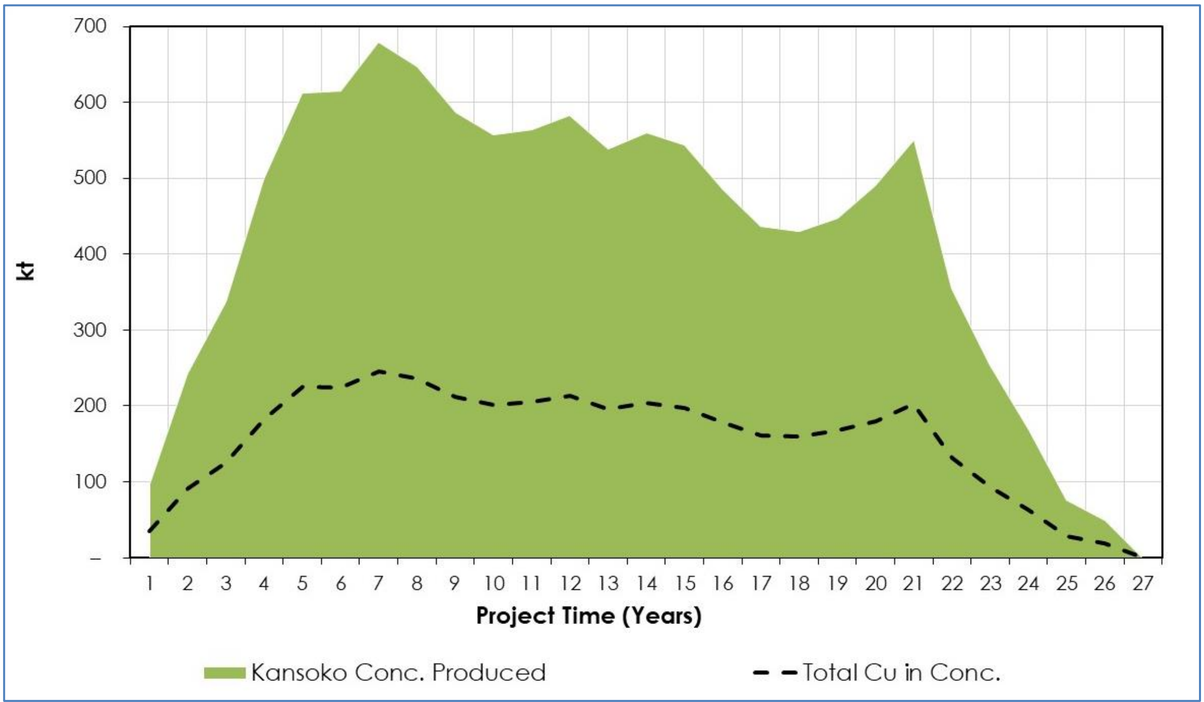


Figure by OreWin, 2016.

Table 22.7 summarises unit operating costs and Table 22.8 provides a breakdown of operating costs and revenue.

Table 22.7 Unit Operating Costs for Kamoa 2017 PFS 6 Mtpa

	US$/lb Payable Cu		
	Years 1–5	Years 1–10	LOM Average
Mine Site	0.62	0.57	0.64
Smelter	–	–	–
Transport	0.47	0.47	0.47
Treatment and Refining Charges	0.18	0.19	0.19
Royalties and Export Tax	0.22	0.22	0.22
Total Cash Costs	1.49	1.44	1.51

Table 22.8 Revenue and Operating Costs

	Total LOM (US$M)	Years 1–5	Years 1–10	LOM Average
		US$/t Milled		
Revenue				
Copper in Concentrate	26,653	238.12	237.40	212.91
Gross Sales Revenue	26,653	238.12	237.40	212.91
Less: Realisation Costs				
Transport	4,142	37.36	37.20	33.08
Treatment and Refining	1,649	14.68	14.73	13.18
Royalties and Export Tax	1,945	17.30	17.36	15.53
Total Realisation Costs	7,736	69.34	69.29	61.79
Net Sales Revenue	18,918	168.77	168.11	151.12
Site Operating Costs				
UG Mining	3,490	31.42	28.47	27.88
Processing	1,469	11.50	11.72	11.74
Tailings	29	0.30	0.23	0.23
General and Administration	774	7.33	6.04	6.18
SNEL Discount	-191	-2.18	-2.19	-1.48
Customs	82	0.73	0.67	0.66
Total	5,654	49.11	44.95	45.21
Operating Margin	13,264	119.66	123.16	105.91
Operating Margin	70.11%	70.90%	73.26%	70.08%

The capital costs for the project are detailed in Table 22.9.

Table 22.9 Capital Investment Summary

Capital Costs (US$M)	Initial Capital US$M	Expansion Capital US$M	Sustaining Capital US$M	Total US$M
Mining				
Underground Mining	311	–	806	1,117
Capitalised Pre-Production	4	–	–	4
Subtotal	315	–	806	1,121
Power				
Power Supply Off Site	71	–	–	71
Capitalised Power Cost	1	–	–	1
Subtotal	72	–	–	72
Concentrator and Tailings				
Plant Capex	146	84	172	402
Tailings	21	95	–	116
Subtotal	167	179	172	518
Infrastructure				
General Infrastructure Capex	110	–	83	193
Other Infrastructure	35	–	26	61
Rail	–	48	–	48
Subtotal	145	48	109	302
Indirects				
EPCM	70	34	21	125
Owners Cost	79	20	8	107
Closure	–	–	76	76
Subtotal	149	54	106	308
Capital Expenditure Before Contingency	848	280	1,193	2,321
Contingency	156	68	141	365
Capital Expenditure After Contingency	1,004	348	1,334	2,686

The cash flow sensitivity to metal price variation is shown in Table 22.10, for copper prices from US$2.00/lb Cu to US$4.00/lb.

The sensitivity of After Tax NPV_8 to initial capital cost, expansion capital cost, direct operating costs, treatment and refining, and transport are shown in Table 22.11. The table shows the change in the base case After Tax NPV_8 of US$2,063 M. The sensitivity to treatment and refining applies the concentrate treatment charges of US$80/t concentrate and concentrate refining charge of US$0.08/lb Cu. The sensitivity to transport applies the costs via road (US$356/t) and via rail (US$323/t).

The change in Cu feed grade is approximately equivalent to a change in recovery or metal price because all three parameters are directly related to copper revenue.

Table 22.10 Metal Price Sensitivity

After Tax NPV (US$M)	Copper Price - US$/lb				
Discount Rate	**2.00**	**2.50**	**3.00**	**3.50**	**4.00**
Undiscounted	1,386	4,417	7,441	10,463	13,479
4.0%	391	2,142	3,874	5,600	7,325
6.0%	102	1,474	2,824	4,167	5,509
8.0%	-102	992	2,063	3,126	4,188
10.0%	-247	640	1,503	2,358	3,212
12.0%	-349	381	1,086	1,783	2,479
15.0%	-447	110	643	1,169	1,693
IRR	6.9%	16.7%	24.2%	30.5%	36.3%

Table 22.11 Additional Sensitivities

			Change from Base NPV_8% (US$M)				
Variable	**Units**	**Base Value**	**-25%**	**-10%**	**–**	**10%**	**25%**
Initial Capital Cost	US$M	1,004	2,236	2,132	2,063	1,993	1,889
Expansion Capital Cost	US$M	348	2,105	2,080	2,063	2,046	2,021
Initial and Expansion Capital Cost	US$M	1,352	2,278	2,149	2,063	1,976	1,847
Site Operating Cost	US$/t Milled	45	2,415	2,205	2,063	1,921	1,707
Treatment and Refining	US$/t and US$/lb Cu	80 / 0.08	2,162	2,103	2,063	2,023	1,964
Transport	US$/t Conc	356 / 323	2,315	2,164	2,063	1,962	1,810

The annual and cumulative cash flows are shown in Figure 22.3 (annual cash flow is shown on the left vertical axis and cumulative cash flow on the right axis). The Project cash flow is shown in Table 22.12.

Figure 22.3 Kamoa 2017 PFS 6 Mtpa Cumulative Cash Flow

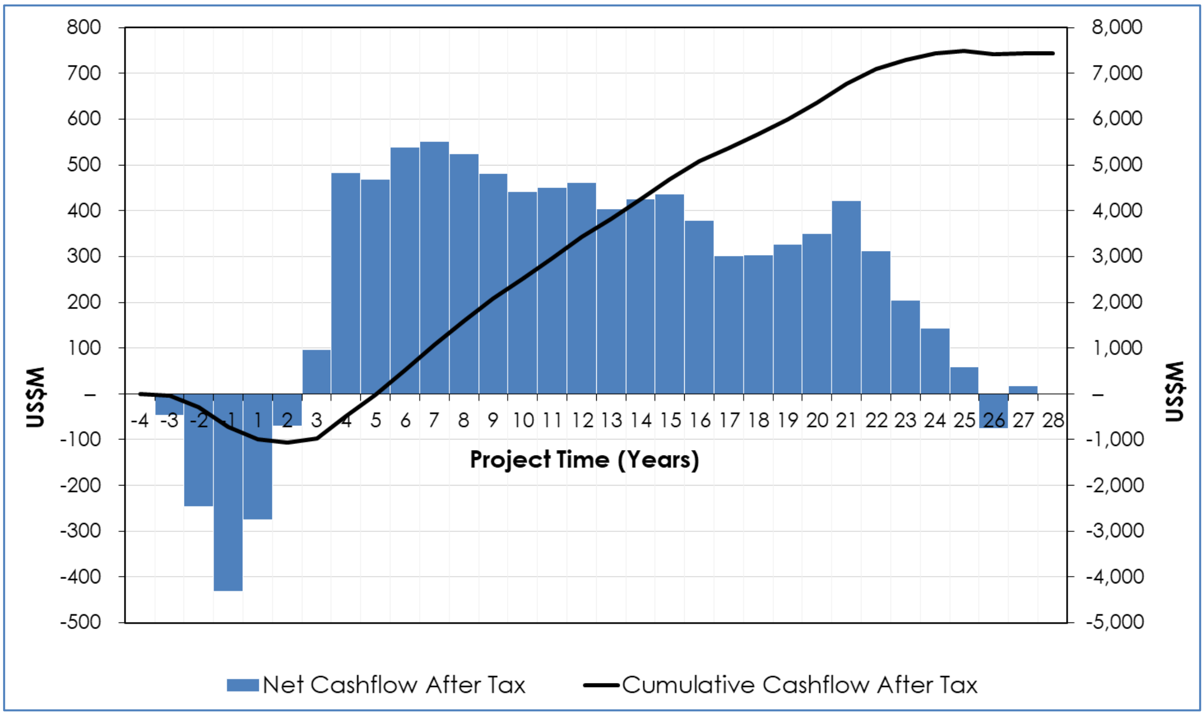


Figure by OreWin, 2017.

Table 22.12 Cash Flow

Cash Flow Statement (US$M)		Year											
Year Number	**Total**	**-4**	**-3**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**11**	**21**
Year To											**10**	**20**	**LOM**
Gross Revenue	26,653	–	–	–	–	226	578	794	1,168	1,441	7,134	11,883	3,430
Realisation Costs	7,736	–	–	–	–	69	174	229	337	415	2,085	3,441	985
Net Revenue	18,918	–	–	–	–	156	405	564	830	1,025	5,049	8,442	2,445
Operating Costs													
Mining	3,490	–	–	–	–	35	71	112	145	192	805	1,651	479
Processing	1,469	–	–	–	–	13	29	36	54	71	357	713	196
Tailings	29	–	–	–	–	1	1	1	1	1	6	12	6
General and Administration	774	–	–	–	–	19	20	29	29	32	159	313	173
Discount on Power	-191	-1	-1	-1	-2	-5	-5	-6	-11	-13	-66	-81	–
Customs (OPEX)	82	–	–	–	–	1	2	3	3	4	19	39	11
Total Operating Costs	5,654	-1	-1	-1	-2	64	118	175	221	288	1,280	2,647	865
Operating Surplus / (Deficit)	13,264	1	1	1	2	92	287	389	609	737	3,769	5,795	1,580
Capital Costs													
Initial Capital	1,004	–	43	226	411	324	–	–	–	–	–	–	–
Expansion Capital	348	–	–	–	–	21	116	84	13	16	46	52	–
Sustaining Capital	1,334	–	–	–	–	–	204	186	94	69	262	339	180
Customs (Capitalised)	66	–	0	2	4	10	9	8	3	3	10	13	3
Working Capital	–	0	4	19	19	12	28	14	15	21	-19	-14	-101
Net Cash Flow Before Tax	10,512	1	-47	-247	-432	-275	-71	96	483	629	3,471	5,406	1,497
Income Tax	3,071	–	–	–	–	–	–	–	–	160	932	1,564	415
Net Cash Flow After Tax	7,441	1	-47	-247	-432	-275	-71	96	483	469	2,539	3,842	1,082

22.5 Capital Cost and Production Benchmarking

Figure 22.4 compares the capital intensity for Large-Scale Copper Projects of Wood Mackenzie's projects currently in construction. The figure shows recently approved projects and other projects rated in the Wood Mackenzie database to be developed with nominal copper production capacity in excess of 200 ktpa. The estimates are based on public disclosure and information gathered by Wood Mackenzie. The Kamoa-Kakula 2017 Development Plan was not reviewed by Wood Mackenzie prior to filing.

Figure 22.4 Capital Intensity for Large-Scale Copper Projects

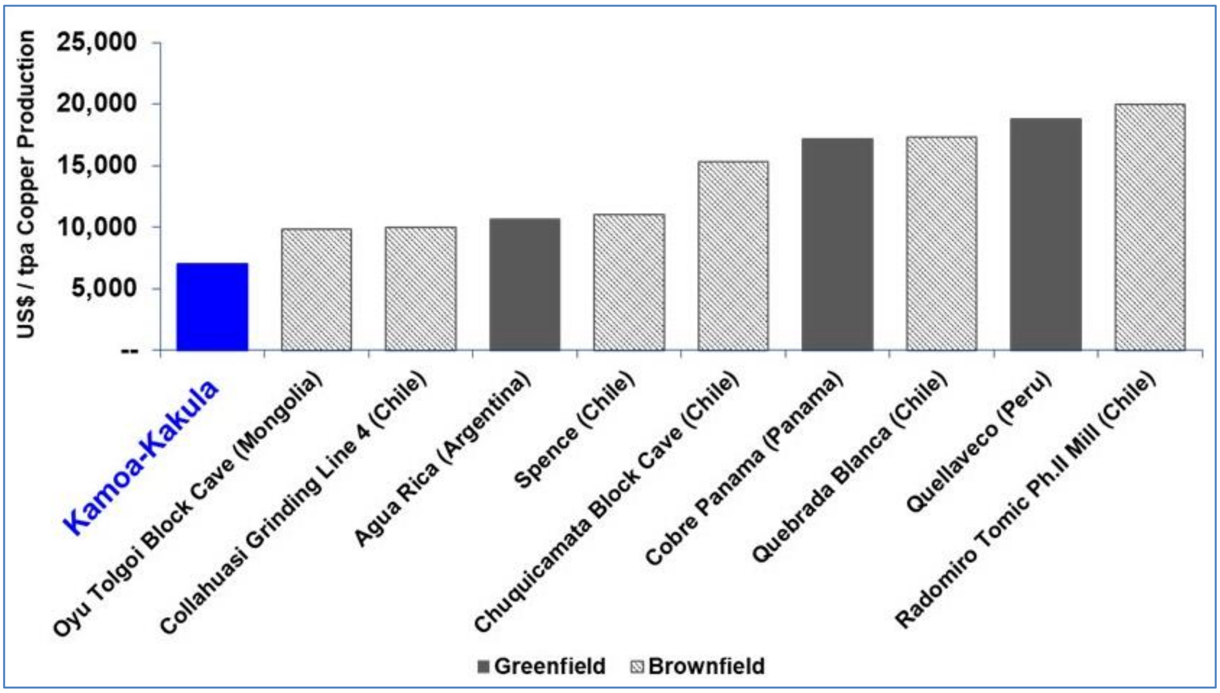


Figure by Ivanhoe, 2017. Source: Wood Mackenzie.

23 ADJACENT PROPERTIES

There are no adjacent properties relevant to this Report.

24 OTHER RELEVANT DATA AND INFORMATION

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

24.1 Kamoa-Kakula 2017 PEA

The Kamoa-Kakula 2017 PEA is part of the Kamoa-Kakula 2017 Development Plan. The Kamoa-Kakula 2017 PEA was prepared to provide two initial scenarios for development of the high-grade copper deposits at the Kamoa-Kakula Project on the Central African Copperbelt, west of the DRC's Katanga mining region.

The Kamoa-Kakula 2017 PEA includes analysis of the Kakula deposit as a standalone operation and an alternative initial option that could involve a two-phase sequential expansion of production to 12 Mtpa from the proposed Kakula Mine, the Kansoko Mine and Kamoa North Mines.

The two PEA production scenarios are:

- Kakula 6 Mtpa PEA (includes the Kakula Mine only).
- Kakula 6 Mtpa, Kansoko 6 Mtpa and Kamoa North 12 Mtpa.

Both the Kamoa-Kakula 2017 PEA scenarios assume initial production from Kakula. The Kakula decline development is followed by the development of the stoping panels and construction of the plant. The Kakula 6 Mtpa PEA initial plant capacity is 3 Mtpa and then expanded to 6 Mtpa.

In the Kamoa-Kakula 12 Mtpa PEA scenario once Kakula reaches full production of 6 Mtpa the Kansoko Mine commences and the plant at Kakula is expanded until the total production rate reaches 12 Mtpa after approximately nine years. The Kamoa-Kakula 12 Mtpa PEA scenario also includes an on-site smelter to produce blister copper at the mine site, which commences production as the 12 Mtpa rate is reached. Once the Kansoko and Kakula Mines near the end of their mine life, Kamoa North comes on line to maintain the overall production rate at 12 Mtpa.

The potential development scenarios at Kamoa-Kakula Project include the 12 Mtpa PEA development scenario is shown in Figure 24.1 and an overview of deposits included within Kamoa-Kakula 2017 PEA (6 Mtpa and 12 Mtpa case) and Kamoa 2017 PFS (6 Mtpa) is shown in Figure 24.2.

Figure 24.1 Kamoa-Kakula 2017 PEA Long-Term Development Plan

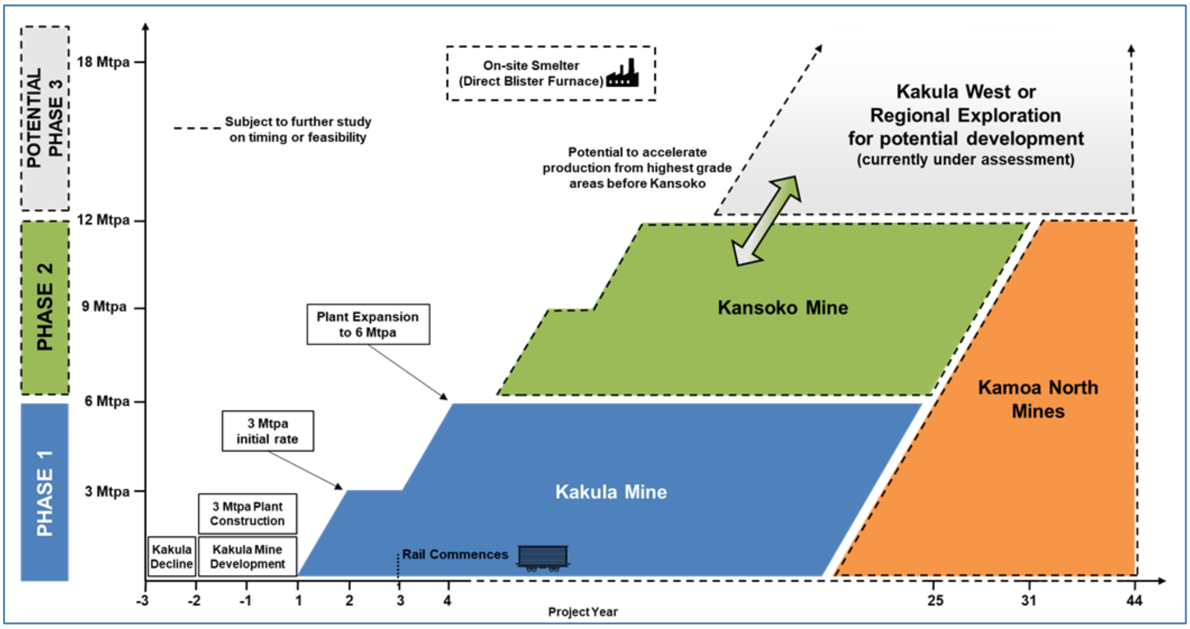


Figure by OreWin, 2017.

Figure 24.2 Overview of Deposits Included Within Kamoa-Kakula 2017 PEA and Kamoa 2017 PFS

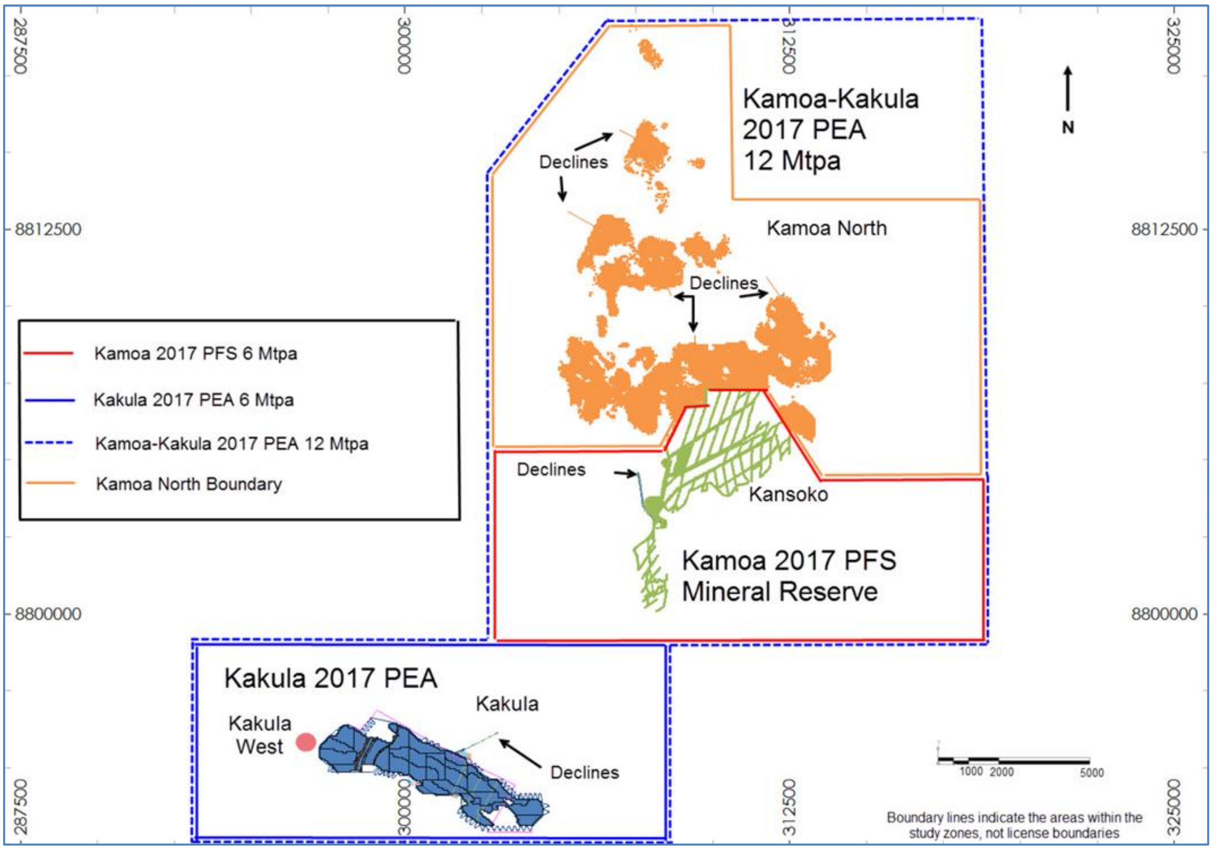


Figure by OreWin, 2017.

The first scenario of the Kamoa-Kakula 2017 PEA, the Kakula 6 Mtpa PEA, represents the initial phase of the Kakula development. This option envisages an average annual production rate of 284 kt of copper at a mine site cash cost of US$0.51/lb copper and total cash cost of US$1.14/lb copper for the first ten years of operations, and annual copper production of up to 320 kt by Year 9. The pre-production capital cost of US$1.2 billion for this option would result in an after-tax net present value at an 8% discount rate (NPV8%) of US$4.2 billion.

The Kamoa-Kakula 2017 PEA is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves – and there is no certainty that the results will be realised. Mineral Resources do not have demonstrated economic viability and are not Mineral Reserves.

The Kamoa-Kakula 12 Mtpa PEA scenario envisages US$1.2 billion in initial capital costs. Future expansion at the Kansoko Mine and subsequent extensions could be funded by cash flows from the Kakula Mine, resulting in an after-tax net present value at an 8% discount rate (NPV8%) of US$7.2 billion and an internal rate of return of 33%. Under this approach, the Kamoa-Kakula 12 Mtpa PEA also includes the construction of a direct-to-blister flash copper smelter with a capacity of 690,000 tonnes of copper concentrate per annum to be funded from internal cash flows. This would be completed in Year five of operations, achieving significant savings in treatment charges and transportation costs.

The Kamoa-Kakula 12 Mtpa PEA scenario has an average annual production of 370,000 tonnes of copper at a total cash cost of US$1.02/lb copper during the first 10 years of operations and production of 542,000 tonnes by Year Nine. At this future production rate, Kamoa-Kakula would rank among the world's five largest copper mines. The results of the two PEA scenarios are summarised in Table 24.1. A plan showing the locations of the mines and key infrastructure for Kakula and Kansoko mines is shown in Figure 24.3.

The Kamoa-Kakula 2017 PEA includes economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability. The results of the Kamoa-Kakula 2017 PEA represent forward looking information. The forward-looking information includes metal price assumptions, cash flow forecasts, projected capital and operating costs, metal recoveries, mine life and production rates, and other assumptions used in the Kakula 2017 Resource Update. Readers are cautioned that actual results may vary from those presented. The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially are presented in the body of this report under each relevant section.

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

Table 24.1 Kamoa-Kakula 2017 PEA Scenarios Summary

Item	Unit	Kakula 6 Mtpa PEA	Kamoa-Kakula 12 Mtpa PEA
Total Processed			
Quantity Milled	kt	108,422	444,276
Copper Feed Grade	%	5.48	3.79
Total Concentrate Produced			
Copper Concentrate Produced	kt (dry)	9,400	34,206
Copper Concentrate - External Smelter	kt (dry)	9,400	9,744
Copper Concentrate - Internal Smelter	kt (dry)	–	24,461
Copper Recovery	%	86.86	85.97
Copper Concentrate Grade	%	54.94	42.30
Cont. Metal in Conc. - External Smelter	Mlb	11,385	10,627
Cont. Metal in Conc. - External Smelter	kt	5,164	4,820
Cont. Metal in Blister - Internal Smelter	Mlb	–	20,955
Cont. Metal in Blister - Internal Smelter	kt	–	9,505
Peak Annual Contained Metal in Concentrate	kt	385	542
10 Year Average			
Copper Feed Grade	%	6.42	5.72
Copper Concentrate Produced	kt (dry)	517	758
Cont. Metal in Conc. - External Smelter	kt	284	188
Cont. Metal in Blister - Internal Smelter	kt	–	182
Mine Site Cash Cost (Including Smelter)	US$/lb	0.51	0.63
Total Cash Cost (After Credits)	US$/lb	1.14	1.02
Key Financial Results			
Peak Funding	US$M	1,135	1,139
Initial Capital Cost	US$M	1,231	1,235
Expansion Capital Cost	US$M	318	3,647
Sustaining Capital Costs	US$M	1,443	5,133
LOM Avg. Mine Site Cash Cost (Including Smelter)	US$/lb	0.60	0.91
LOM Avg. Total Cash Costs (After Credits)	US$/lb	1.23	1.20
Site Operating Cost	US$/t Milled	61.49	64.17
After-Tax NPV8%	US$M	4,243	7,179
After-Tax IRR	%	36.2	33.0
Project Payback	Years	3.1	4.7
Initial Project Life	Years	24	44

Figure 24.3 Planned Kamoa-Kakula 2017 PEA Site Plan

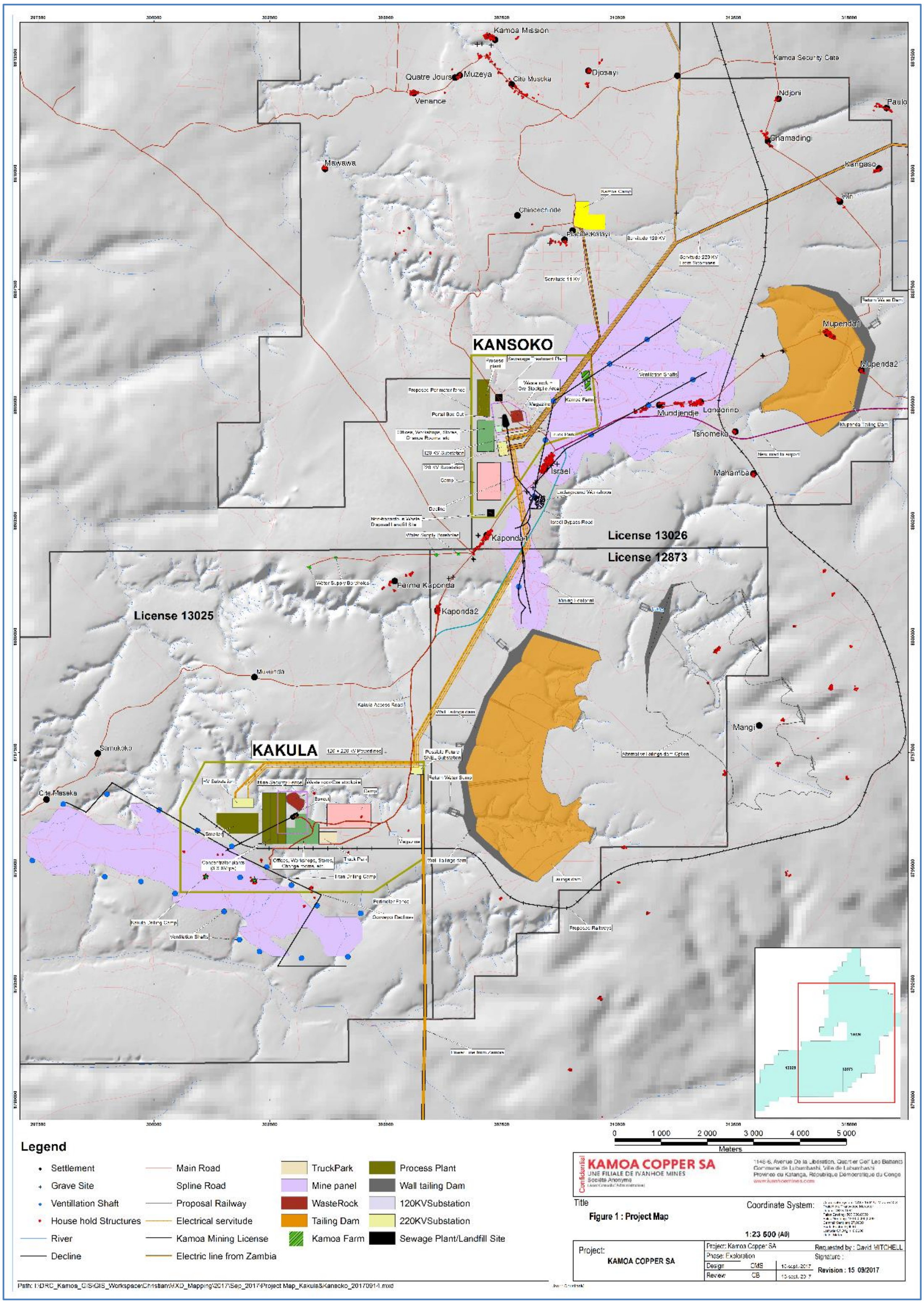


Figure by Ivanhoe, 2017.

The Kamoa-Kakula 12 Mtpa PEA development scenario and long-term development plan is shown in Figure 24.4.

Figure 24.4 8 Mtpa Development Scenario

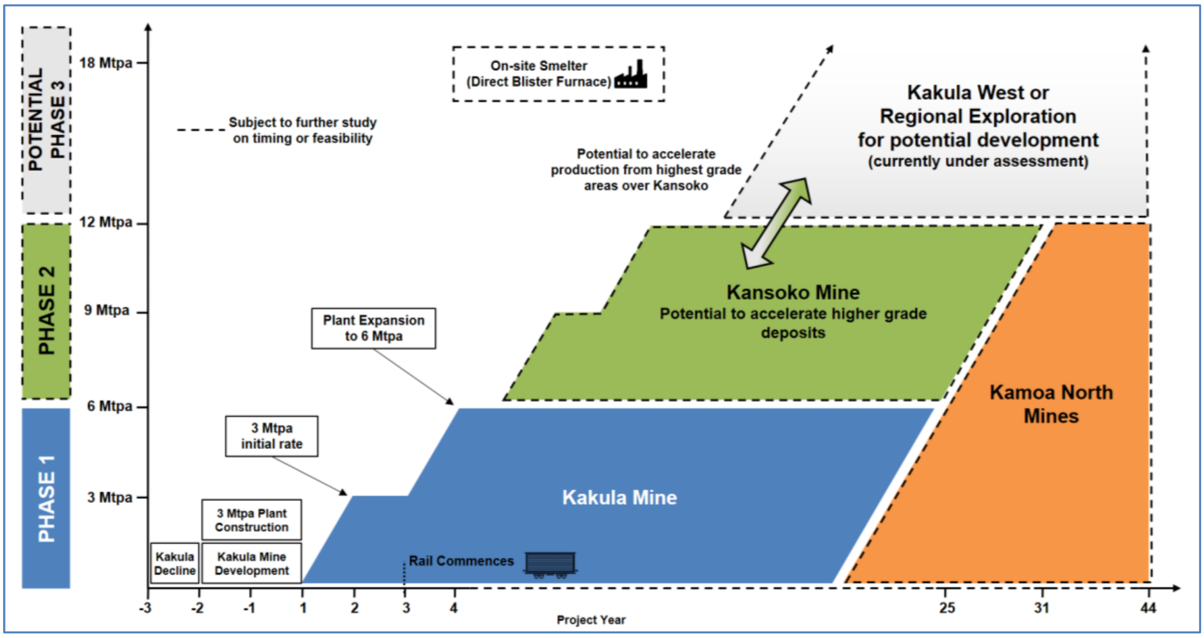


Figure by OreWin, 2017.

Based on initial metallurgical testwork, the chalcocite-rich nature of the copper mineralisation at the Kakula deposit is expected to yield higher metallurgical recoveries and higher concentrate grades, which in turn are expected to reduce unit transportation costs and therefore improve financial returns.

24.2 Kakula 2017 PEA Assumptions

Estimates of cash flows have been prepared on a real basis as at 1 December 2017 and discounted to a Net Present Value (NPV) at a rate of 8% for all years ($NPV_{8\%}$). The NPV results have been calculated starting from December 2017 and a mid-year discounting approach is taken.

The key economic assumptions for the analyses are shown in Table 24.2. Copper price was selected based on reviews of long term consensus estimates and metal prices reported in public reports. The discount rate of 8% was selected for the base case after a review of public reporting for base metal projects. Smelter terms for treatment and refining charges were selected from published long-term contract terms and forecasts.

Table 24.2 Kakula 2017 PEA Financial Analysis Assumptions

Parameter	Unit	Financial Analysis Assumptions
Copper Price	US$/lb	3.00
Copper Treatment Charge	US$/dmt conc.	80.00
Copper Refining Charge	US$/lb Cu	0.08

24.3 Kamoa-Kakula 2017 PEA Results Summary

24.3.1 Kakula 2017 PEA 6 Mtpa Scenario Results Summary

The Kakula 6 Mtpa PEA represents the initial phase of the Kakula development. The Kakula 2017 PEA evaluates the development of a 6 Mtpa underground mine and surface processing complex at the Kakula Deposit – a discovery announced in early 2016 – as the project's first phase of development. The development scenario of the Kakula Mine on the Kakula Deposit is shown in Figure 24.5.

This PEA analyses the potential development of an initial 6 Mtpa Kakula Mine at the Kakula Deposit in the southerly portion of the Kamoa-Kakula Project's discovery area. For this option, the PEA envisages an average annual production rate of 284 kt of copper at a mine site cash cost of US$0.51/lb copper and total cash cost of US$1.14/lb copper for the first ten years of operations, and copper annual production of up to 320 kt by Year 9. The pre-production capital cost of US$1.2 billion for this option would result in an after-tax net present value at an 8% discount rate (NPV8%) of US$4.2 billion.

Figure 24.5 Kakula 2017 PEA 6 Mtpa Development Scenario

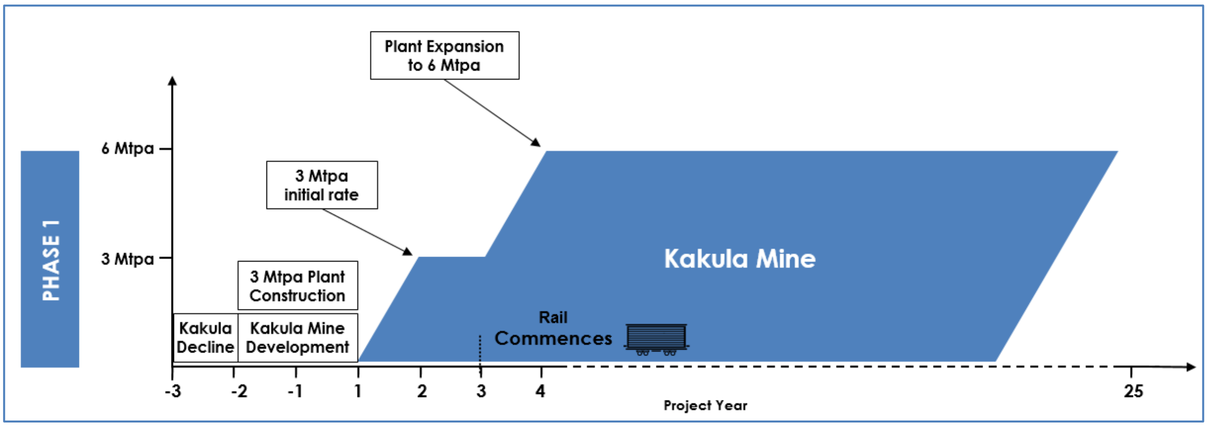


Figure by OreWin, 2017.

A summary of the key results for the Kakula 2017 PEA 6 Mtpa scenario are:

- Very-high-grade initial phase of production is projected to have a grade of 7.3% copper in year four and an average grade of 6.4% copper over the initial 10 years of operations, resulting in estimated average annual copper production of 284,000 tonnes.
- Annual copper production is estimated at 385,000 tonnes in year four.
- Initial capital cost, including contingency, is estimated at US$1.2 billion.
- Average total cash cost of US$1.14/lb of copper during the first 10 years.
- After-tax NPV, at an 8% discount rate, of US$4.2 billion.
- After-tax internal rate of return (IRR) of 36.2%, and a payback period of 3.1 years.
- Kakula is expected to produce a very-high-grade copper concentrate in excess of 50% copper, with extremely low arsenic levels.

The study assesses the potential development of the Kakula Deposit as a 6 Mtpa mining and processing complex. The Kakula mill would be constructed in two smaller phases of 3 Mtpa each as the mining operations ramp-up to full production of 6 Mtpa. The life-of-mine production scenario provides for 108.4 million tonnes to be mined at an average grade of 5.48% copper, producing 9.4 million tonnes of high-grade copper concentrate, containing approximately 11.4 billion pounds of copper.

The economic analysis uses a long-term price assumption of US$3.00/lb of copper and returns an after-tax NPV at an 8% discount rate of US$4.2 billion. It has an after-tax IRR of 36.2% and a payback period of 3.1 years.

The estimated initial capital cost, including contingency, is US$1.2 billion. The capital expenditure for off-site power, which is included in the initial capital cost, includes a US$71 million advance payment to the DRC state-owned electricity company, SNEL, to upgrade two hydropower plants (Koni and Mwadingusha) to provide the Kamoa-Kakula Project with access to clean electricity for its planned operations. Mwadingusha is being upgraded first. The work is being led by Stucky Ltd., of Switzerland; the advance payment will be recovered through a reduction in the power tariff.

The Kakula 2017 PEA is preliminary in nature and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically for the application of economic considerations that would allow them to be categorised as Mineral Reserves – and there is no certainty that the results will be realised. Mineral Resources do not have demonstrated economic viability and are not Mineral Reserves. The results of the Kamoa-Kakula 2017 Development Plan represent forward looking information. The forward-looking information includes metal price assumptions, cash flow forecasts, projected capital and operating costs, metal recoveries, mine life and production rates, and other assumptions used in the Kakula 2017 Resource Update. Readers are cautioned that actual results may vary from those presented. The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially are presented in the body of this report under each relevant section.

Key results of the Kakula 2017 PEA for a single 6 Mtpa mine are summarised in Table 24.3.

Table 24.3 Kakula Mine Results Summary for 6 Mtpa Production

Item	Unit	Total
Total Processed		
Quantity Milled	kt	108,422
Copper Feed Grade	%	5.48
Total Concentrate Produced		
Copper Concentrate Produced	kt (dry)	9,400
Copper Recovery	%	86.86
Copper Concentrate Grade	%	54.94
Contained Metal in Concentrate	Mlb	11,385
Contained Metal in Concentrate	kt	5,164
Peak Annual Contained Metal in Concentrate	kt	385
10-Year Average		
Copper Concentrate Produced	kt (dry)	517
Contained Metal in Concentrate	kt	284
Mine-Site Cash Cost	US$/lb	0.51
Total Cash Cost	US$/lb	1.14
5-Year Average		
Copper Concentrate Produced	kt (dry)	448
Contained Metal in Concentrate	kt	246
Mine-Site Cash Cost	US$/lb	0.45
Total Cash Cost	US$/lb	1.08
Key Financial Results		
Peak Funding	US$M	1,135
Initial Capital Cost	US$M	1,231
Expansion Capital Cost	US$M	318
Sustaining Capital Costs	US$M	1,443
LOM Average Mine Site Cash Cost	US$/lb Cu	0.60
LOM Average Total Cash Cost	US$/lb Cu	1.23
Site Operating Cost	US$/t Milled	61.49
After-Tax NPV8%	US$M	4,243
After-Tax IRR	%	36.2
Project Payback Period	Years	3.1
Initial Project Life	Years	24

Table 24.4 summarizes the financial results. The mining production statistics are shown in Table 24.5. The Kakula 2017 PEA 6 Mtpa mill feed and copper grade profile for the first 20 years are shown in Figure 24.6 and the concentrate and metal production for the first 20 years are shown in Figure 24.7.

Table 24.4 Kakula Mine Financial Results for 6 Mtpa Production

Net Present Value (US$M)	Discount Rate	Before Taxation	After Taxation
	Undiscounted	16,607	11,700
	4.0%	9,940	6,919
	6.0%	7,816	5,398
	8.0%	6,200	4,243
	10.0%	4,955	3,353
	12.0%	3,984	2,660
Internal Rate of Return	–	43.0%	36.2%
Project Payback Period (Years)	–	2.9	3.1

Table 24.5 Kakula Mine Average Estimated Production and Processing Statistics for 6 Mtpa Production

Item	Unit	Years 1-5	Years 1-10	LOM Average
Total Processed				
Quantity Milled	kt	4,135	5,073	4,518
Copper Feed Grade	%	6.80	6.42	5.48
Annual Concentrate Produced				
Copper Concentrate Produced	kt (dry)	448	517	392
Copper Recovery	%	87.46	87.29	86.86
Copper Concentrate Grade	%	54.94	54.94	54.94
Contained Metal in Concentrate				
Copper	Mlb	543	627	474
Copper	kt	246	284	215
Payable Metal				
Copper	Mlb	530	612	463
Copper	kt	240	277	210

Figure 24.6 Kakula Mine Estimated Tonnes Milled and Head Grade for the First 20 Years

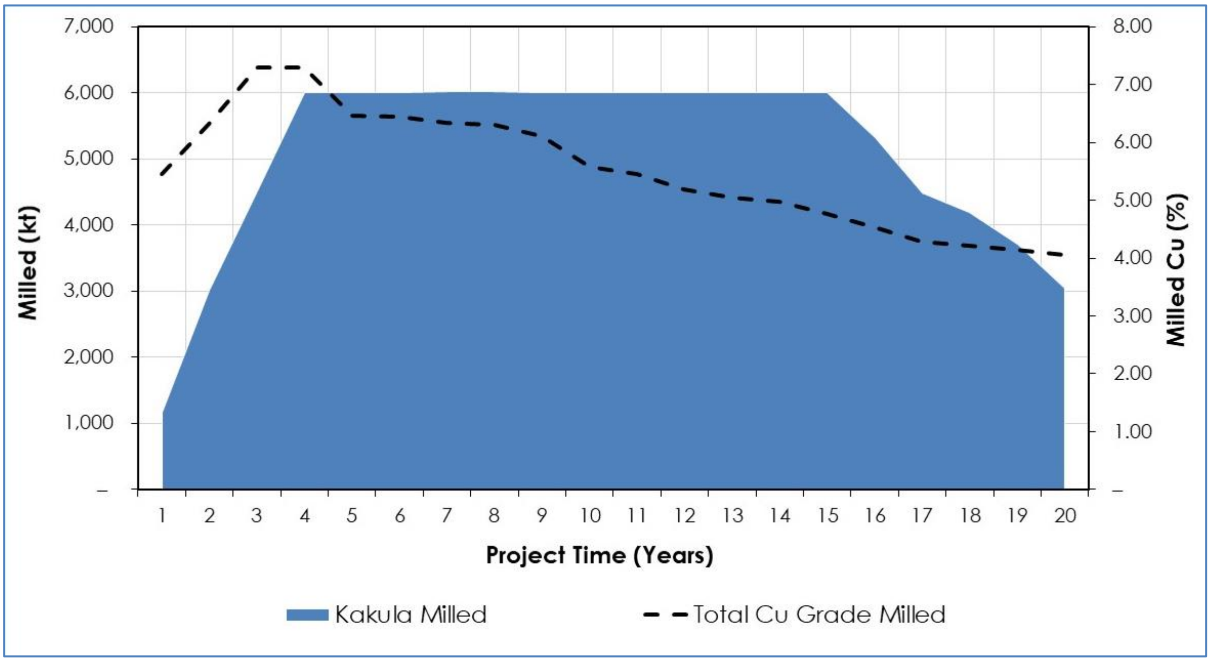


Figure by OreWin, 2017.

Figure 24.7 Kakula Mine Estimated Concentrate and Metal Production for the First 20 Years

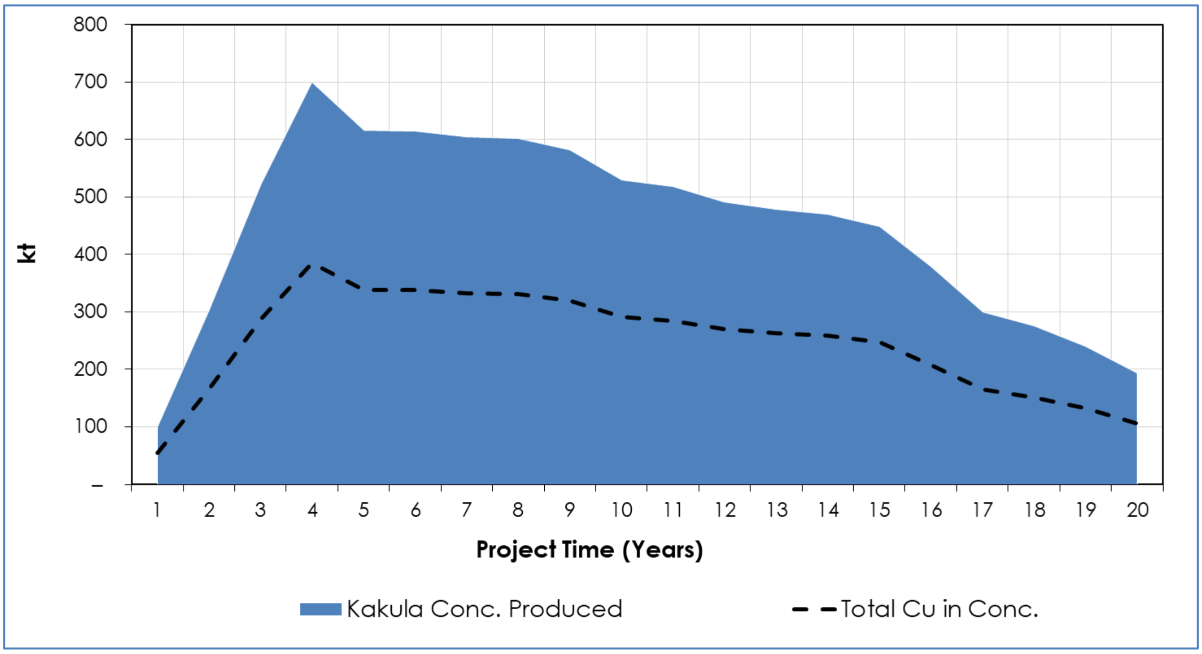


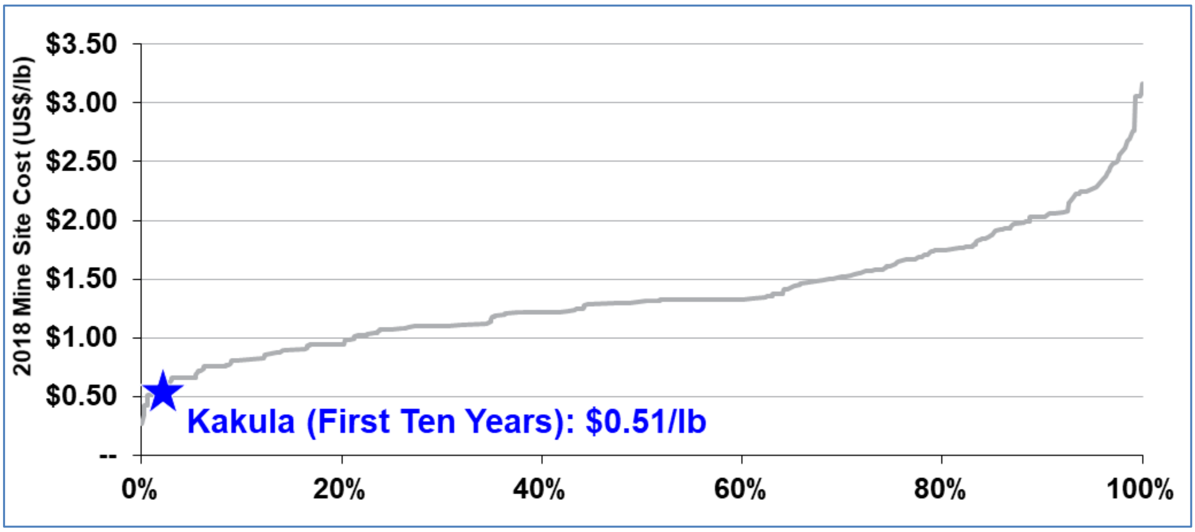
Figure by OreWin, 2017.

Table 24.6 summarizes unit operating costs. Figure 24.8 compares the average mine-site cash cost during the first 10 years of the Kakula 2017 PEA and Wood Mackenzie's comparable projects and Figure 24.9 compares the C1 pro-rata copper cash costs of the Kakula 2017 PEA and Wood Mackenzie's comparable projects.

Table 24.6 Kakula Mine Unit Operating Costs for 6 Mtpa Production

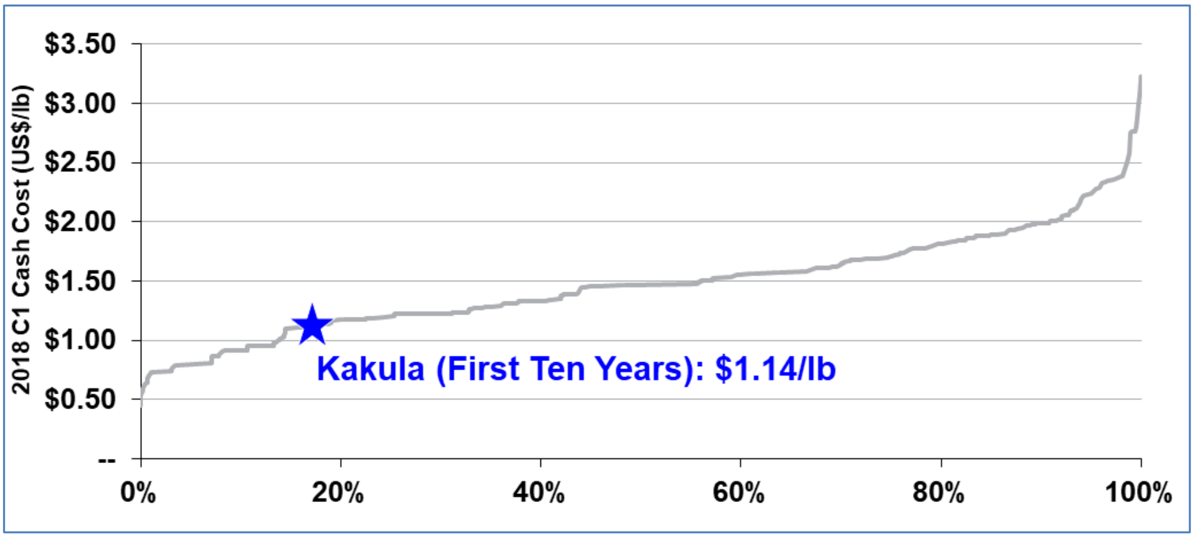
	US$/lb Payable Copper		
	Years 1-5	Years 1-10	LOM Average
Mine Site	0.45	0.51	0.60
Transport	0.31	0.31	0.31
Treatment and Refining Charges	0.15	0.15	0.15
Royalties and Export Tax	0.17	0.17	0.17
Total Cash Costs	1.08	1.14	1.23

Figure 24.8 2018 Mine-Site Cash Costs (Includes All Operational Costs at Mine Site)



Note: Represents mine-site cash costs that reflect the direct cash costs of producing paid concentrate or cathode incorporating mining, processing and mine-site G&A costs. Kakula is based on the average mine-site cash cost during the first 10 years as detailed in the Kakula 2017 PEA. Source: Wood Mackenzie (based on public disclosure, the Kakula 2017 PEA has not been reviewed by Wood Mackenzie).

Figure 24.9 2018 C1 Pro-Rata Copper Cash Costs (includes mining, processing, transportation and off-site realization costs)



Note: Represents C1 pro-rata cash costs that reflect the direct cash costs of producing paid metal incorporating mining, processing, mine-site G&A and offsite realization costs, having made appropriate allowance for the costs associated with the co-product revenue streams. Kakula is based on the average total cash cost during the first 10 years as detailed in the Kakula 2017 PEA. Source: Wood Mackenzie (based on public disclosure, the Kakula 2017 PEA has not been reviewed by Wood Mackenzie).

Table 24.7 provides a breakdown of revenue and operating costs. Capital costs for the project are detailed in Table 24.8.

Table 24.7 Kakula Mine Estimated Revenue and Operating Costs For 6 Mtpa Production

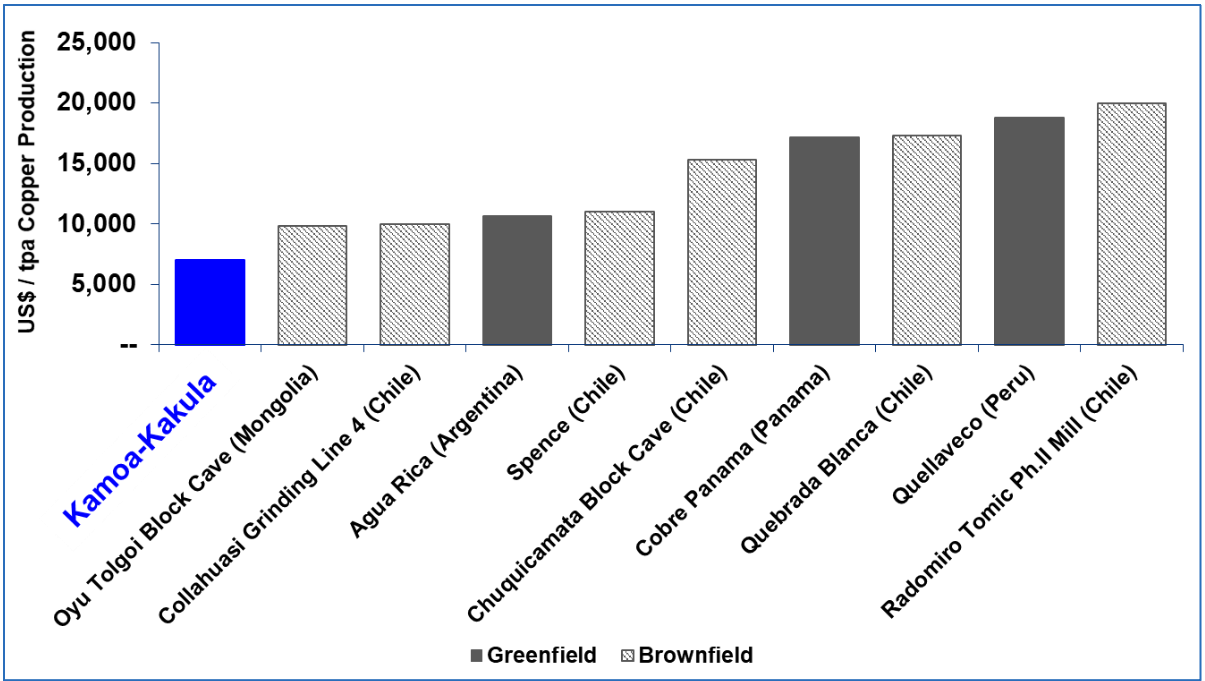
	Total LOM (US$M)	Years 1–5	Years 1–10	LOM Average
		US$/t Milled		
Revenue				
Copper in Concentrate	33,346	384.31	361.76	307.56
Gross Sales Revenue	33,346	384.31	361.76	307.56
Less: Realization Costs				
Transport	3,418	39.93	37.21	31.52
Treatment and Refining	1,663	19.16	18.04	15.34
Royalties and Export Tax	1,935	22.29	20.99	17.85
Total Realization Costs	7,015	81.38	76.24	64.70
Net Sales Revenue	26,331	302.93	285.53	242.86
Site Operating Costs				
Underground Mining	4,679	39.94	44.65	43.16
Processing	1,308	12.00	12.14	12.06
Tailings	29	0.30	0.25	0.26
General and Administration	728	6.36	5.77	6.71
SNEL Discount	-187	-2.12	-2.23	-1.67
Customs	104	0.91	0.99	0.96
Total	6,661	57.38	61.57	61.49
Net Operating Margin	19,670	245.55	223.96	181.37
Net Operating Margin	74.70%	81.06%	78.44%	74.68%

Table 24.8 Kakula 6 Mtpa PEA Capital Costs

Description	Initial Capital (US$M)	Expansion Capital (US$M)	Sustaining Capital (US$M)	Total (US$M)
Mining				
Underground Mining	403	–	1,045	1,447
Capitalised Pre-Production	36	–	–	36
Subtotal	438	–	1,045	1,483
Power				
Power Supply Off Site	71	–	–	71
Capitalised Power Cost	4	–	–	4
Subtotal	75	–	–	75
Concentrate and Tailings				
Process Plant	146	84	159	389
Tailings	27	74	–	101
Subtotal	173	158	159	489
Infrastructure				
Mine Surface Infrastructure	35	–	24	59
General Infrastructure	110	–	76	187
Rail Link	–	48	–	48
Subtotal	145	48	100	293
Indirects				
EPCM	78	31	–	109
Owners Cost	95	20	–	115
Closure	–	–	75	75
Subtotal	173	51	75	298
Capital Expenditure Before Contingency	1,004	257	1,378	2,638
Contingency	227	62	65	354
Capital Expenditure After Contingency	1,231	318	1,443	2,992

Figure 24.10 compares the capital intensity for Large-Scale Copper Projects of Wood Mackenzie's projects currently in construction.

Figure 24.10 Capital Intensity for Large-Scale Copper Projects



Note: Recently approved, probable and possible projects with nominal copper production capacity in excess of 200 ktpa (based on public disclosure and information gathered in the process of routine research). The Kakula 2017 PEA has not been reviewed by Wood Mackenzie. Source: Wood Mackenzie.

The after-tax NPV sensitivity to metal price variation is shown in Table 24.9 for copper prices from US$2.00/lb to US$4.00/lb. The annual and cumulative cash flows are shown in Figure 24.11 (annual cash flow is shown on the left vertical axis and cumulative cash flow on the right axis).

Table 24.9 Kakula Mine Copper Price Sensitivity

After-Tax NPV (US$M)	Copper Price - US$/lb				
Discount Rate	**2.00**	**2.50**	**3.00**	**3.50**	**4.00**
Undiscounted	4,135	7,921	11,700	15,478	19,253
4.0%	2,257	4,591	6,919	9,247	11,573
6.0%	1,654	3,529	5,398	7,267	9,135
8.0%	1,195	2,722	4,243	5,764	7,282
10.0%	841	2,100	3,353	4,606	5,856
12.0%	567	1,617	2,660	3,703	4,744
IRR	18.9%	28.6%	36.2%	42.8%	48.6%

Figure 24.11 Kakula Mine Projected Cumulative Cash Flow

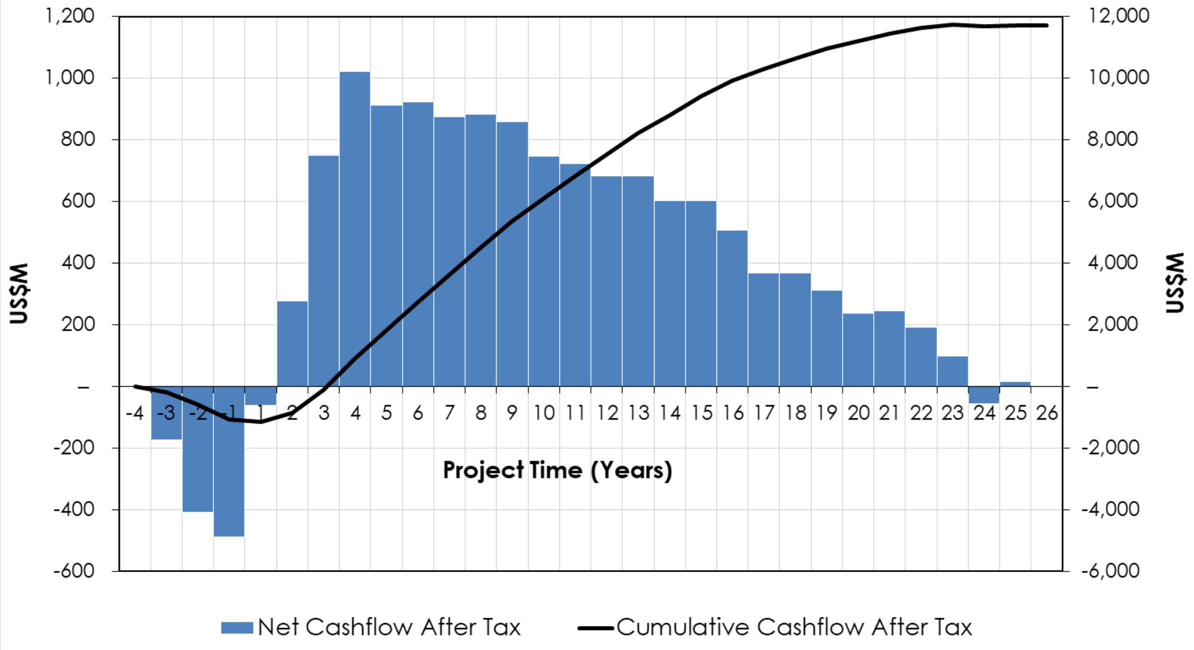


Figure by OreWin, 2017.

Table 24.10 Cash Flow – Kakula 6 Mtpa PEA

Cash Flow Statement (US$M)		Year											
Year Number	**Total**	**-4**	**-3**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**11**	**21**
Year To											**10**	**20**	**LOM**
Gross Revenue	33,346	–	–	–	–	357	1,069	1,850	2,484	2,186	10,406	13,467	1,527
Realisation Costs	7,015	–	–	–	–	78	235	388	521	459	2,185	2,827	321
Net Revenue	26,331	–	–	–	–	278	834	1,461	1,962	1,727	8,222	10,640	1,207
Operating Costs													
Mining	4,679	–	–	–	–	62	113	142	230	278	1,439	2,155	259
Processing	1,308	–	–	–	–	13	35	53	74	73	368	612	80
Tailings	29	–	–	–	–	1	1	1	1	1	6	12	4
General and Administration	728	–	–	–	–	19	20	30	30	33	161	316	118
Discount on Power	-187	-1	-1	-1	-2	-5	-6	-6	-13	-14	-69	-68	–
Customs (OPEX)	104	–	–	–	–	1	3	4	5	6	32	48	6
Total Operating Costs	6,661	-1	-1	-1	-2	92	166	223	327	378	1,937	3,077	467
Operating Surplus / (Deficit)	19,670	1	1	1	2	186	668	1,238	1,635	1,349	6,285	7,563	740
Capital Costs													
Initial Capital	1,231	5	155	382	475	213	–	–	–	–	–	–	–
Expansion Capital	318	–	–	–	–	21	118	97	–	13	49	20	–
Sustaining Capital	1,443	–	–	–	–	–	188	102	118	93	296	509	136
Customs (Capex)	71	0	2	4	5	7	9	6	4	3	11	17	2
Working Capital	–	1	16	24	10	7	75	66	51	-31	-32	-119	-67
Net Cash Flow Before Tax	16,607	-4	-172	-409	-488	-62	277	968	1,462	1,271	5,960	7,136	668
Income Tax	4,907	–	–	–	–	–	–	218	440	358	1,674	2,048	169
Net Cash Flow After Tax	11,700	-4	-172	-409	-488	-62	277	750	1,022	913	4,286	5,088	499

Table 24.11 Processing Production Schedule – Kakula 6 Mtpa PEA

Description	Units	Totals	Project Time (Years)														
			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Quantity Milled	kt	108,422	1,169	3,006	4,484	6,010	6,007	6,010	6,013	6,018	6,012	6,002	6,009	6,004	6,007	6,009	6,000
Cu Feed Grade	% Cu	5.48	5.45	6.31	7.29	7.30	6.46	6.44	6.34	6.30	6.10	5.57	5.46	5.19	5.05	4.97	4.76
Copper Conc. Produced	kt (dry)	9,400	101	301	521	700	616	615	605	602	582	529	518	491	478	470	449
Copper Concentrate Recovery	%	86.86	86.80	87.23	87.65	87.65	87.30	87.29	87.24	87.23	87.14	86.87	86.81	86.66	86.57	86.53	86.39
Copper Concentrate Grade	% Cu	54.94	54.94	54.94	54.94	54.94	54.94	54.94	54.94	54.94	54.94	54.94	54.94	54.94	54.94	54.94	54.94
Total Recovered Copper Production	Mlb	11,385	122	365	632	848	746	745	733	729	705	641	628	595	579	570	544
Total Recovered Copper Production	kt	5,164	55	166	286	385	339	338	332	331	320	291	285	270	263	258	247

Description	Units	Project Time (Years)															
		16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31
Quantity Milled	kt	5,326	4,482	4,184	3,718	3,048	3,020	2,350	1,045	492	–	–	–	–	–	–	–
Cu Feed Grade	% Cu	4.54	4.28	4.21	4.14	4.05	4.02	4.01	3.92	3.81	–	–	–	–	–	–	–
Copper Conc. Produced	kt (dry)	379	300	276	241	193	190	147	64	29	–	–	–	–	–	–	–
Copper Concentrate Recovery	%	86.24	86.06	86.00	85.95	85.88	85.86	85.85	85.77	85.68	–	–	–	–	–	–	–
Copper Concentrate Grade	% Cu	54.94	54.94	54.94	54.94	54.94	54.94	54.94	54.94	54.94	–	–	–	–	–	–	–
Total Recovered Copper Production	Mlb	459	364	334	292	234	230	178	78	35	–	–	–	–	–	–	–
Total Recovered Copper Production	kt	208	165	151	132	106	104	81	35	16	–	–	–	–	–	–	–

24.3.2 Kamoa-Kakula 12 Mtpa PEA Results Summary

The Kamoa-Kakula 12 Mtpa PEA assesses the development of both the Kakula and Kamoa deposits as an integrated, 12 Mtpa mining and processing complex. Each operation is expected to be a separate underground mine with associated dedicated processing facilities and surface infrastructure. The Kamoa-Kakula 12 Mtpa PEA scenario envisages the construction and operation of two separate facilities: the Kakula Mine on the Kakula Deposit and the Kansoko Mine on the Kansoko Sud and Kansoko Centrale areas of the Kamoa Deposit. The Kakula Mine scenario is the same as that presented in the Kakula 2017 PEA 6 Mtpa. The initial plant capacity of 3 Mtpa is expanded to 6 Mtpa as the Kansoko Mine and Kakula Mine ramp up. The mines continue to ramp up to 12 Mtpa combined by Year 9. Once the Kansoko and Kakula Mines near the end of their mine life, Kamoa North comes on line to maintain the overall production at 12 Mtpa. The 12 Mtpa PEA also analyses an on-site smelter to produce blister copper at the mine site.

The Kamoa-Kakula 12 Mtpa PEA envisages US$1.2 billion in initial capital costs. Future expansion at the Kansoko Mine and subsequent extensions could be funded by cash flows from the Kakula Mine, resulting in an after-tax net present value at an 8% discount rate (NPV8%) of US$7.2 billion and an internal rate of return of 33%. Under this approach, the 12 Mtpa PEA also includes the construction of a direct-to-blister flash copper smelter with a capacity of 690,000 tonnes of copper concentrate per annum to be funded from internal cash flows. This would be completed in Year Five of operations, achieving significant savings in treatment charges and transportation costs.

The Kamoa-Kakula 12 Mtpa PEA scenario has an average annual production of 370,000 tonnes of copper at a total cash cost of US$1.02/lb copper during the first 10 years of operations and production of 542,000 tonnes by Year Nine. At this future production rate, Kamoa-Kakula would rank among the world's five largest copper mines.

A summary of the key results for the Kamoa-Kakula 12 Mtpa PEA scenario are:

- Very-high-grade initial phase projected to have a grade of 7.3% copper in Year Four and an average grade of 5.72% copper during the first 10 years of operations, resulting in estimated average annual copper production of 370,000 tonnes.
- Annual copper production is estimated at 542,000 tonnes in Year Nine, ranking Kamoa-Kakula as potentially one of the five largest copper mines in the world.
- Initial capital cost, including contingency, is US$1.2 billion, with subsequent expansions from Kansoko and other mining areas, as well as the smelter, to be funded by cash flows from the Kakula Mine.
- Average total cash costs of US$1.02/lb of copper during the first 10 years, including sulphuric acid credits.
- After-tax NPV, at an 8% discount rate, of US$7.2 billion.
- After-tax IRR of 33% and a payback period of 4.7 years.

The Kamoa-Kakula 12 Mtpa PEA development scenario and long-term development plan is shown in Figure 24.12. Key results of the Kamoa-Kakula 12 Mtpa PEA Scenario are summarised in Table 24.12. The production results for the external smelter (concentrate sales to off site customers) and internal smelter (on site smelter owned by the project) scenarios are both shown in Table 24.13.

The Kamoa-Kakula 2018 Resource Update includes economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorised as Mineral Reserves, and there is no certainty that the results will be realised. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability. The results of the Kamoa-Kakula 2018 Resource Update represent forward looking information. The forward-looking information includes metal price assumptions, cash flow forecasts, projected capital and operating costs, metal recoveries, mine life and production rates, and other assumptions used in the Kamoa-Kakula 2018 Resource Update. Readers are cautioned that actual results may vary from those presented. The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially are presented in the body of this report under each relevant section.

Figure 24.12 Kamoa-Kakula 2017 PEA Long-Term Development Plan

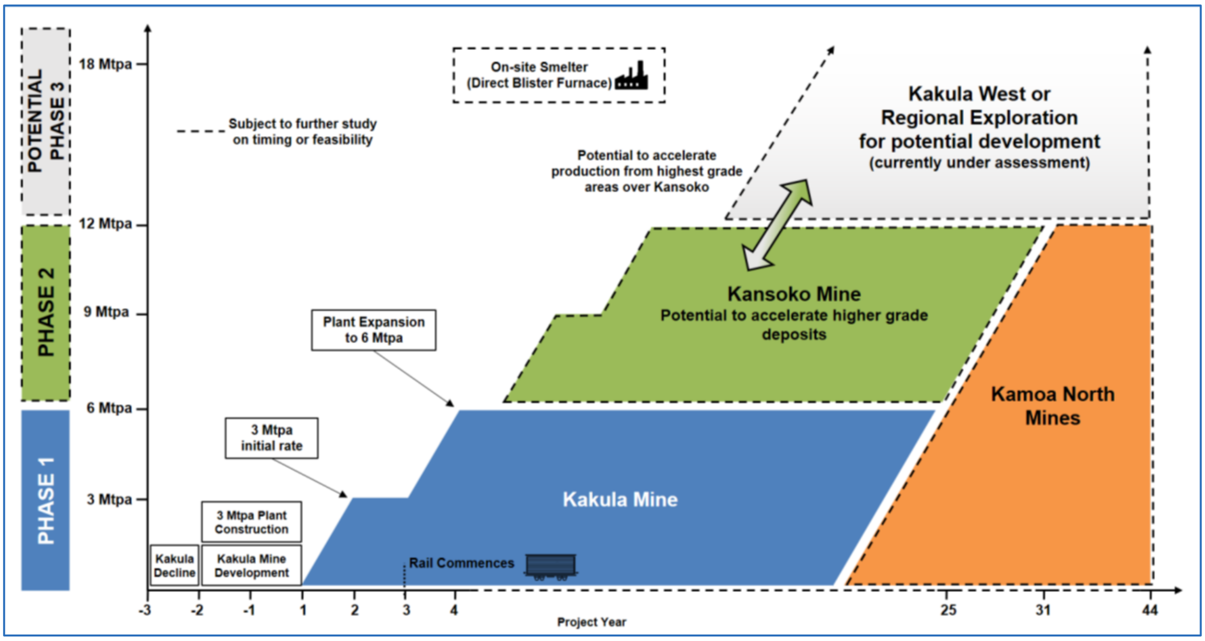


Figure by OreWin, 2017.

Table 24.12 Results Summary – Kamoa-Kakula 12 Mtpa PEA

Item	Unit	Total
Total Processed		
Quantity Milled	kt	444,276
Copper Feed Grade	%	3.79
Total Concentrate Produced		
Copper Concentrate Produced	kt (dry)	34,206
Copper Concentrate - External Smelter	kt (dry)	9,744
Copper Concentrate - Internal Smelter	kt (dry)	24,461
Copper Recovery	%	85.97
Copper Concentrate Grade	%	42.30
Cont. Metal in Conc. - External Smelter	Mlb	10,627
Cont. Metal in Conc. - External Smelter	kt	4,820
Cont. Metal in Blister - Internal Smelter	Mlb	20,955
Cont. Metal in Blister - Internal Smelter	kt	9,505
Peak Annual Contained Metal in Concentrate	kt	542
10 Year Average		
Copper Feed Grade	%	5.72
Copper Concentrate Produced	kt (dry)	758
Cont. Metal in Conc. - External Smelter	kt	188
Cont. Metal in Blister - Internal Smelter	kt	182
Mine Site Cash Cost (Including Smelter)	US$/lb	0.63
Total Cash Cost (After Credits)	US$/lb	1.02
Key Financial Results		
Peak Funding	US$M	1,139
Initial Capital Cost	US$M	1,235
Expansion Capital Cost	US$M	3,647
Sustaining Capital Costs	US$M	5,133
LOM Avg. Mine Site Cash Cost (Including Smelter)	US$/lb	0.91
LOM Avg. Total Cash Costs (After Credits)	US$/lb	1.20
Site Operating Cost	US$/t Milled	64.17
After-Tax NPV8%	US$M	7,179
After-Tax IRR	%	33.0
Project Payback	Years	4.7
Initial Project Life	Years	44

Table 24.13 Kakula 12 Mtpa PEA Production and Processing

Item	Unit	Total LOM	Years 1-5	Years 1-10	LOM Aeerage.
Total Processed					
Quantity Milled	kt	444,276	4,369	7,442	10,097
Copper Feed Grade	%	3.79	6.63	5.72	3.79
Concentrate Produced					
Copper Concentrate Produced	kt (dry)	34,206	467	758	777
Copper Concentrate - External Smelter	kt (dry)	9,744	329	344	221
Copper Concentrate - Internal Smelter	kt (dry)	24,461	138	414	556
Copper Recovery	%	85.97	87.47	87.51	85.97
Copper Concentrate Grade	%	42.30	54.17	49.18	42.30
Contained Metal in Concentrate - External Smelter					
Copper	Mlb	10,627	399	415	242
Copper	kt	4,820	181	188	110
Payable Metal in Concentrate - External Smelter					
Copper	Mlb	10,348	390	405	235
Copper	kt	4,694	177	184	107
Contained Metal in Blister - Internal Smelter					
Copper	Mlb	20,955	157	400	476
Copper	kt	9,505	71	182	216
Payable Metal in Blister - Internal Smelter					
Copper	Mlb	20,892	156	399	475
Copper	kt	9,476	71	181	215
Payable Metal					
Copper	Mlb	31,240	546	804	710
Copper	kt	14,170	248	365	322

Table 24.14 summarizes unit operating costs and Table 24.15 summarizes the financial results.

Table 24.14 Unit Operating Costs for Kamoa-Kakula 12 Mtpa PEA

	US$/lb Payable Copper		
	Years 1–5	Years 1–10	LOM Average
Mine Site (ex-Smelter)	0.46	0.54	0.78
Smelter	0.05	0.09	0.13
Transport	0.27	0.23	0.21
Treatment and Refining Charges	0.12	0.10	0.09
Royalties and Export Tax	0.15	0.13	0.12
Total Cash Costs Before Credits	1.04	1.09	1.33
Sulphuric Acid Credits[1]	(0.03)	(0.07)	(0.13)
Total Cash Costs After Credits	1.02	1.02	1.20

Assumes a sulphuric acid price of US$200 per tonne.

Table 24.15 Kakula Mine Financial Results for Kamoa-Kakula 12 Mtpa PEA

Net Present Value (US$M)	Discount Rate	Before Taxation	After Taxation
	Undiscounted	45,948	31,970
	4.0%	20,784	14,283
	6.0%	14,704	10,008
	8.0%	10,676	7,179
	10.0%	7,916	5,243
	12.0%	5,966	3,879
Internal Rate of Return		40.0%	33.0%
Project Payback Period (Years)		3.7	4.7

The 12 Mtpa PEA mill feed and copper grade profile are shown in Figure 24.13 and the concentrate and metal production are shown in Figure 24.14.

Figure 24.13 12 Mtpa PEA Scenario Mill Feed and Grade Profile

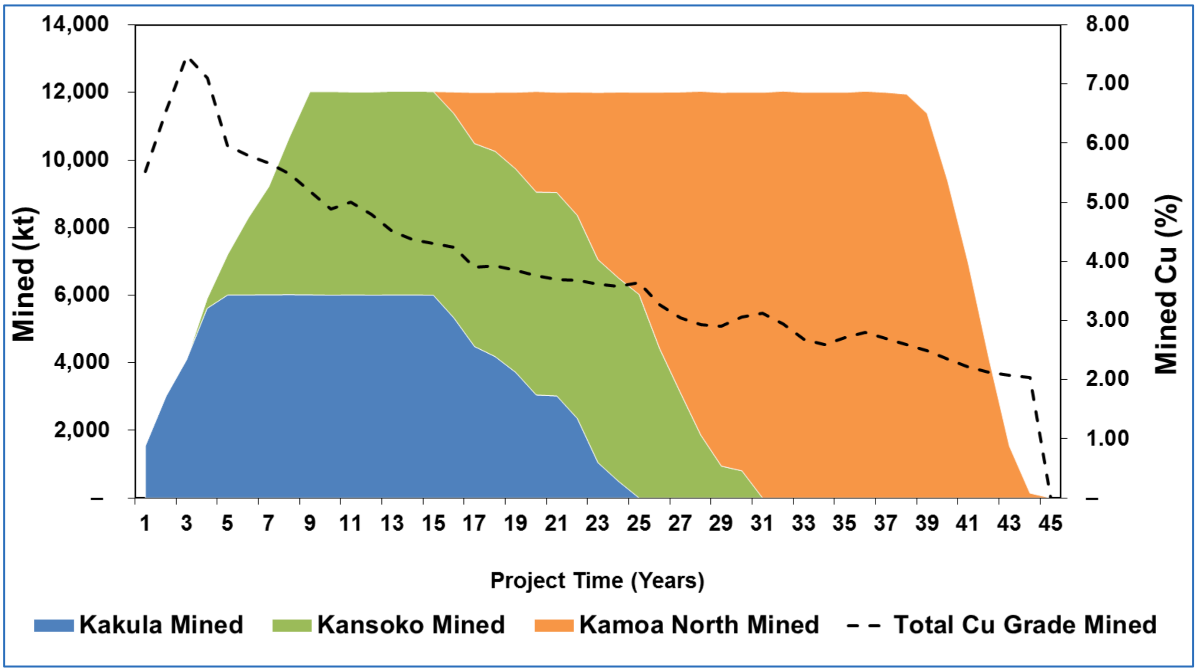


Figure by OreWin, 2017.

Figure 24.14 12 Mtpa PEA Scenario Concentrate and Metal Production

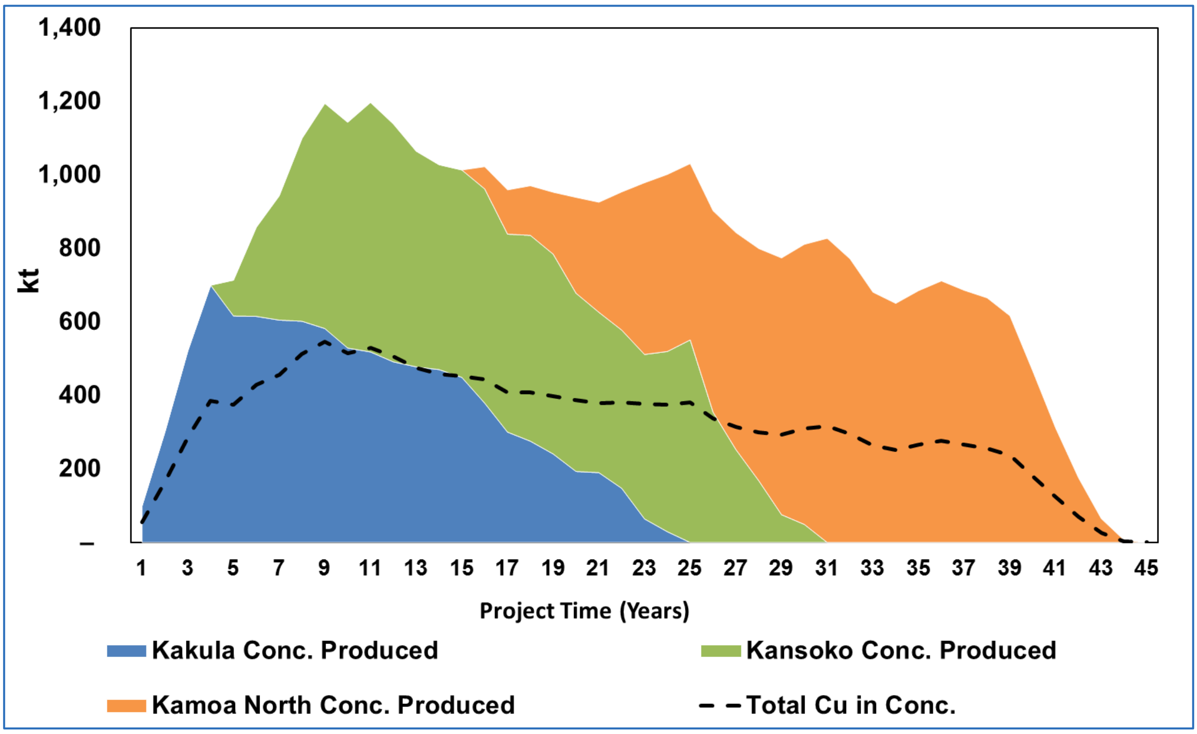


Figure by OreWin, 2017.

The revenue summary is shown in Table 24.16. The revenue streams include for the smelter case include payable copper contained within concentrate, payable copper in blister and acid production. A copper price of US$3.00 /lb and an acid selling price of US$200 /t are assumed. Table 24.17 provides a breakdown of revenue and operating costs. Capital costs for the project are detailed in Table 24.18.

Table 24.16 Gross Revenue Summary

Description	Revenue (US$M)	Gross Revenue (%)
Payable Copper in Blister	62,676	64%
Payable Copper in Concentrate	31,044	32%
Acid Production	4,053	4%
Gross Revenue	**97,773**	**100%**

Table 24.17 Estimated Revenue and Operating Costs for 12 Mtpa Production

	Total LOM (US$M)	Years 1–5	Years 1–10	LOM Average
		US$/t Milled		
Revenue				
Copper in Concentrate	97,773	378.06	332.10	220.07
Gross Sales Revenue	97,773	378.06	332.10	220.07
Less: Realisation Costs				
Transport	6,662	33.31	24.92	14.99
Treatment and Refining	2,675	15.13	10.84	6.02
Royalties and Export Tax	3,689	18.57	14.07	8.30
Total Realisation Costs	13,027	67.01	49.83	29.32
Net Sales Revenue	84,746	311.05	282.27	190.75
Site Operating Costs				
UG Mining	16,042	39.90	40.80	36.11
Processing	6,209	11.62	13.26	13.98
Tailings	60	0.32	0.19	0.13
Smelter	4,021	6.53	9.94	9.05
General and Administration	2,005	6.94	5.27	4.51
SNEL Discount	-171	-2.63	-2.22	-0.37
Customs	337	0.87	0.82	0.76
Total	28,503	63.54	68.05	64.17
Operating Margin	56,243	247.51	214.21	126.58
Operating Margin	66.37%	79.57%	75.89%	66.36%

Table 24.18 Estimated Capital Investment Summary for 12 Mtpa Production

Capital Costs (US$M)	Initial Capital US$M	Expansion Capital US$M	Sustaining Capital US$M	Total
Mining				
Underground Mining	403	1,292	2,981	4,676
Capitalised Pre-Production	36	–	–	36
Subtotal	438	1,292	2,981	4,712
Power and Smelter				
Direct Blister Furnace incl slag handling	–	564	727	1,291
Power Infrastructure	–	–	–	–
Power Supply Off Site	71	–	–	71
Capitalised Power Cost	4	–	–	4
Subtotal	75	564	727	1,366
Concentrator and Tailings				
Plant Capex	146	318	599	1,063
Tailings	29	157	–	186
Subtotal	175	475	599	1,249
Infrastructure				
General Infrastructure Capex	110	106	280	497
Additional Infrastructure Costs (Indirects and Directs)	–	3	4	7
Other Infrastructure	35	9	56	99
Rail	–	48	–	48
Subtotal	145	166	339	650
Indirects				
EPCM	78	278	–	356
Owners Cost	95	196	–	291
Closure	–	–	207	207
Subtotal	173	474	207	854
Capital Expenditure Before Contingency	1,007	2,970	4,854	8,831
Contingency	228	677	279	1,183
Capital Expenditure After Contingency	1,235	3,647	5,133	10,015

The after-tax NPV sensitivity to metal price variation is shown in Table 24.19 for copper prices from US$2.00/lb to US$4.00/lb. The annual and cumulative cash flows are shown in Figure 24.15 (annual cash flow is shown on the left vertical axis and cumulative cash flow on the right axis).

Table 24.19 Copper Price Sensitivity for Kamoa-Kakula 12 Mtpa PEA

After-Tax NPV (US$M)	Copper Price (US$/lb)				
Discount Rate	**2.00**	**2.50**	**3.00**	**3.50**	**4.00**
Undiscounted	10,638	21,313	31,970	42,598	53,213
4.0%	4,540	9,414	14,283	19,146	24,005
6.0%	2,969	6,492	10,008	13,522	17,033
8.0%	1,913	4,549	7,179	9,808	12,435
10.0%	1,187	3,218	5,243	7,267	9,290
12.0%	679	2,282	3,879	5,475	7,069
IRR	16.6%	25.5%	33.0%	39.6%	45.5%

Figure 24.15 Kamoa-Kakula 12 Mtpa PEA Cash Flow

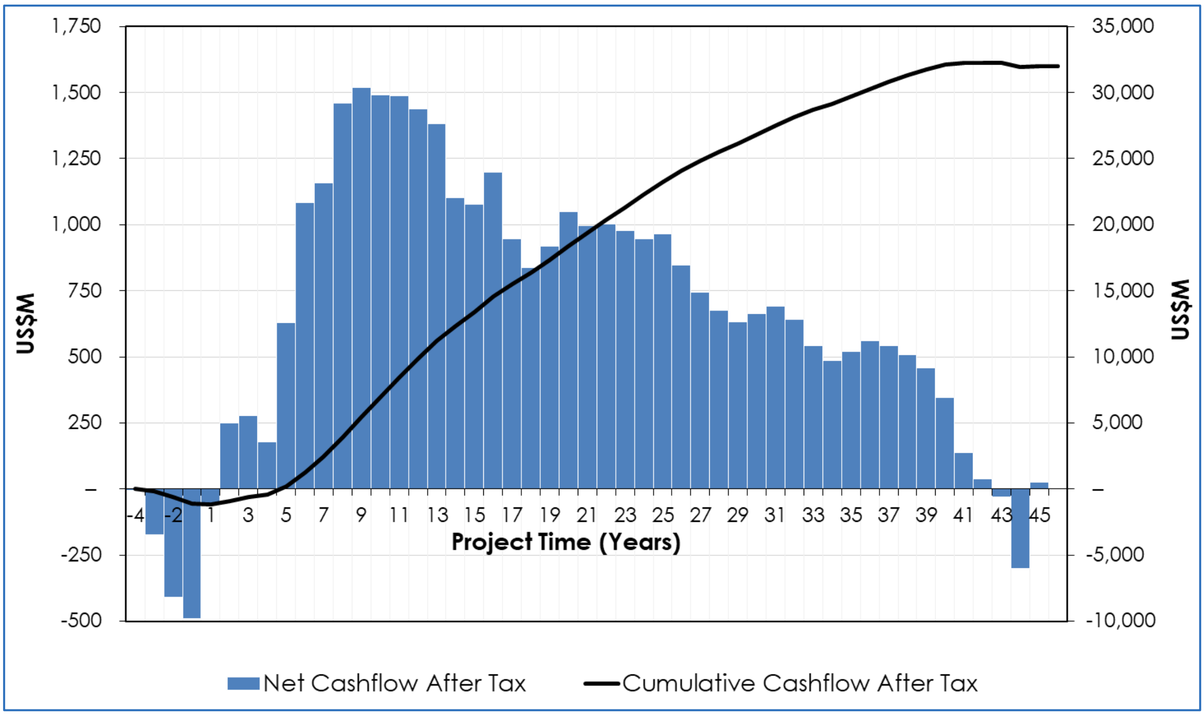


Figure by OreWin, 2017.

Table 24.20 Cash Flow – Kakula 12 Mtpa PEA

Cash Flow Statement (US$M)		Year											
Year Number	**Total**	**-4**	**-3**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**11**	**21**
Year To											**10**	**20**	**LOM**
Gross Revenue	97,773	–	–	–	–	357	1,069	1,850	2,484	2,499	16,457	30,105	42,953
Realisation Costs	13,027	–	–	–	–	78	235	388	521	240	2,245	4,153	5,165
Net Revenue	84,746	–	–	–	–	278	834	1,461	1,962	2,259	14,212	25,951	37,788
Operating Costs													
Mining	16,042	–	–	–	–	62	113	142	234	319	2,165	4,232	8,773
Processing	6,209	–	–	–	–	13	32	49	69	91	733	1,699	3,523
Tailings	60	–	–	–	–	1	1	1	1	1	7	14	32
Smelter	4,021	–	–	–	–	–	–	–	–	143	597	1,099	2,183
General and Administration	2,005	–	–	–	–	19	20	30	39	44	241	513	1,100
Discount on Power	-171	-1	-1	-1	-2	-5	-6	-6	-13	-27	-108	–	–
Customs (OPEX)	337	–	–	–	–	1	3	3	5	6	42	81	195
Total Operating Costs	28,503	-1	-1	-1	-2	91	164	220	336	578	3,677	7,637	15,806
Operating Surplus / (Deficit)	56,243	1	1	1	2	187	670	1,242	1,626	1,681	10,535	18,314	21,982
Capital Costs													
Initial Capital	1,235	5	155	384	476	215	–	–	–	–	–	–	–
Expansion Capital	3,647	–	–	–	–	21	115	503	804	549	602	1,053	–
Sustaining Capital	5,133	–	–	–	–	–	216	130	147	122	601	1,186	2,729
Customs (Capex)	281	0	2	4	5	7	10	18	26	19	37	70	82
Working Capital	–	1	16	24	10	7	78	108	92	-6	18	-88	-259
Net Cash Flow Before Tax	45,948	-4	-172	-411	-489	-63	250	483	557	996	9,277	16,094	19,430
Income Tax	13,978	–	–	–	–	–	–	205	378	366	2,567	4,659	5,804
Net Cash Flow After Tax	31,970	-4	-172	-411	-489	-63	250	278	179	631	6,710	11,435	13,626

Table 24.21 Processing Production Schedule – Kakula 12 Mtpa PEA

Description	Units	Totals	Project Time (Years)														
			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Quantity Milled	kt	444,276	1,169	3,006	4,484	6,010	7,176	8,618	9,218	10,685	12,026	12,030	12,018	12,018	12,034	12,036	12,028
Cu Feed Grade	% Cu	3.79	5.45	6.31	7.29	7.30	5.97	5.70	5.67	5.48	5.18	4.89	5.00	4.79	4.51	4.36	4.31
Copper Conc. Produced	kt (dry)	34,206	101	301	521	700	714	858	943	1,100	1,195	1,143	1,197	1,139	1,065	1,028	1,014
Copper Conc. - External Smelter	kt (dry)	9,744	101	301	521	700	24	168	253	410	505	453	507	449	375	338	324
Copper Conc. -Internal Smelter	kt (dry)	24,461	–	–	–	–	690	690	690	690	690	690	690	690	690	690	690
Copper Concentrate Recovery	%	85.97	86.80	87.23	87.65	87.65	87.34	87.24	87.48	87.72	87.56	87.60	88.13	87.88	87.52	87.51	87.27
Copper Concentrate Grade	% Cu	42.30	54.94	54.94	54.94	54.94	52.40	49.97	48.44	46.71	45.66	45.05	44.27	44.42	44.59	44.69	44.59
Contained Copper in Conc. - External Smelter	Mlb	10,627	122	365	632	848	29	203	306	496	608	539	571	519	449	409	392
Contained Copper in Conc. - External Smelter	kt	4,820	55	166	286	385	13	92	139	225	276	245	259	235	204	186	178
Contained Copper in Blister - Internal Smelter	Mlb	20,955	–	–	–	–	784	731	690	627	586	587	588	588	588	594	595
Contained Copper in Blister - Internal Smelter	kt	9,505	–	–	–	–	356	331	313	284	266	266	267	267	267	270	270
Total Recovered Copper Production	Mlb	31,582	122	365	632	848	813	934	997	1,123	1,194	1,127	1,160	1,106	1,038	1,004	987
Total Recovered Copper Production	kt	14,325	55	166	286	385	369	424	452	509	542	511	526	502	471	455	448

Description	Units	Project Time (Years)															
		16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31
Quantity Milled	kt	12,014	11,998	12,003	12,007	12,028	12,007	12,009	12,000	12,010	12,007	12,007	12,013	12,034	12,001	12,007	12,003
Cu Feed Grade	% Cu	4.23	3.90	3.93	3.84	3.75	3.69	3.68	3.61	3.58	3.64	3.26	3.05	2.93	2.90	3.06	3.12
Copper Conc. Produced	kt (dry)	1,023	959	971	953	939	926	954	979	1,002	1,031	903	843	800	774	811	828
Copper Conc. - External Smelter	kt (dry)	333	269	281	263	249	236	264	289	312	341	213	153	110	84	121	138
Copper Conc. -Internal Smelter	kt (dry)	690	690	690	690	690	690	690	690	690	690	690	690	690	690	690	690
Copper Concentrate Recovery	%	87.20	87.02	86.57	86.36	85.67	85.59	86.29	86.74	87.20	87.10	85.79	85.80	85.07	84.08	84.35	84.37
Copper Concentrate Grade	% Cu	43.36	42.44	42.07	41.82	41.18	41.00	40.00	38.41	37.46	36.92	37.26	37.28	37.52	37.86	38.18	38.21
Contained Copper in Conc. - External Smelter	Mlb	381	302	306	290	273	253	273	262	265	277	175	126	91	70	102	116
Contained Copper in Conc. - External Smelter	kt	173	137	139	132	124	115	124	119	120	126	79	57	41	32	46	53
Contained Copper in Blister - Internal Smelter	Mlb	587	587	585	580	571	575	560	558	553	553	558	559	562	567	572	573
Contained Copper in Blister - Internal Smelter	kt	266	266	265	263	259	261	254	253	251	251	253	253	255	257	259	260
Total Recovered Copper Production	Mlb	968	889	891	870	844	828	833	820	819	831	733	685	653	638	674	688
Total Recovered Copper Production	kt	439	403	404	395	383	376	378	372	371	377	332	311	296	289	306	312

Description	Units	Project Time (Years)															
		32	33	34	35	36	37	38	39	40	41	42	43	44	45	46	47
Quantity Milled	kt	12,037	12,004	12,006	12,003	12,037	12,003	11,951	11,384	9,381	6,949	4,147	1,532	142	–	–	–
Cu Feed Grade	% Cu	2.95	2.68	2.59	2.71	2.79	2.69	2.59	2.49	2.35	2.22	2.13	2.07	2.03	–	–	–
Copper Conc. Produced	kt (dry)	771	681	651	685	712	686	666	617	466	311	176	65	6	–	–	–
Copper Conc. - External Smelter	kt (dry)	81	–	–	–	22	–	–	–	–	311	176	65	6	–	–	–
Copper Conc. -Internal Smelter	kt (dry)	690	681	651	685	690	686	666	617	466	–	–	–	–	–	–	–
Copper Concentrate Recovery	%	83.41	81.89	81.31	81.89	82.33	82.44	82.85	83.16	81.96	80.12	79.52	80.14	81.43	–	–	–
Copper Concentrate Grade	% Cu	38.33	38.65	38.81	38.81	38.90	38.86	38.56	38.26	38.75	39.68	39.87	39.22	37.80	–	–	–
Contained Copper in Conc. - External Smelter	Mlb	69	–	–	–	19	–	–	–	–	272	155	56	5	–	–	–
Contained Copper in Conc. - External Smelter	kt	31	–	–	–	8	–	–	–	–	124	70	25	2	–	–	–
Contained Copper in Blister - Internal Smelter	Mlb	574	572	548	577	583	579	557	512	392	–	–	–	–	–	–	–
Contained Copper in Blister - Internal Smelter	kt	261	259	249	262	264	263	253	232	178	–	–	–	–	–	–	–
Total Recovered Copper Production	Mlb	643	572	548	577	601	579	557	512	392	272	155	56	5	–	–	–
Total Recovered Copper Production	kt	292	259	249	262	273	263	253	232	178	124	70	25	2	–	–	–

24.4 Kakula 2017 PEA Production

24.4.1 Kakula 6 Mtpa PEA Mine Schedule

Mining was scheduled to preferentially extract the higher-grade primary panels before the lower-grade secondary panels. Access development within the panels was scheduled by adjusting the panel access based on length and applying the necessary lag time for completion. Decline development rates were scheduled as per Table 24.22.

Upon completion of the decline, initial access development and ventilation, mining was scheduled at a rate of 6 Mtpa after a four year ramp up period. Scheduling assumed the access development single heading jumbo rate was 160 m/month. After the ramp up, the total decline development rate was assumed to be 320 m/month utilising two jumbos and multiple headings. A rate of 3 m/day was used for ventilation raise bore vertical development. Figure 24.17 shows the development metres over the life of the project.

The mining rate used for scheduling the room-and-pillar with hydraulic fill panels was 205 ktpa and the rate used for scheduling controlled convergence room-and-pillar panels was 722 ktpa.

Kakula mining production is shown in Figure 24.16 and Table 24.23. Over the Kakula mine life, 65% of the overall tonnes are mined using drift-and-fill, 28% are mined using controlled convergence room-and-pillar, with the remaining 7% being from development.

Figure 24.18 shows the Kakula 6 Mtpa PEA quantity of material mined split by resource category.

Table 24.22 Kakula 6 Mtpa PEA Mine Schedule Decline Development

Description	Unit	Total	Project Month Number											
			1	2	3	4	5	6	7	8	9	10	11	12
			Dec 17	Jan 18	Feb 18	Mar 18	Apr 18	May 18	Jun 18	Jul 18	Aug 18	Sep 18	Oct 18	Nov 18
Conveyor Decline "CD" (7W x 6H)	m	1,500	30	100	120	150	125	150	125	150	125	150	125	150
Service Decline "SD" (5.5W x 6H)	m	1,500	30	120	100	120	150	120	150	120	150	120	150	170
Laterals & Muck Bays (5.5W x 6H)	m	400	–	–	43	43	43	43	43	43	43	43	43	13
Cubbies & Sumps (5.5W x 6H)	m	153	–	–	17	17	17	17	17	17	17	17	17	–
Total	m	3,553	60	220	280	330	335	330	335	330	335	330	335	333

Figure 24.16 Kakula 6 Mtpa PEA Production

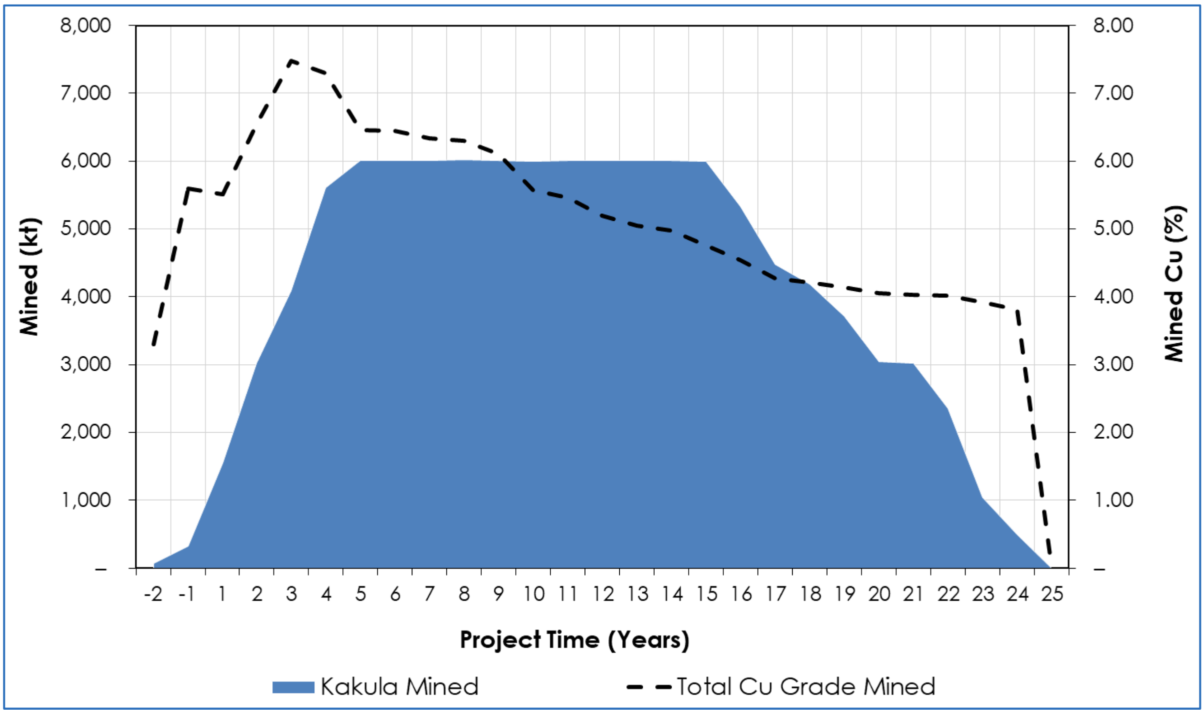


Figure by OreWin, 2017.

Figure 24.17 Kakula 6 Mtpa PEA Development

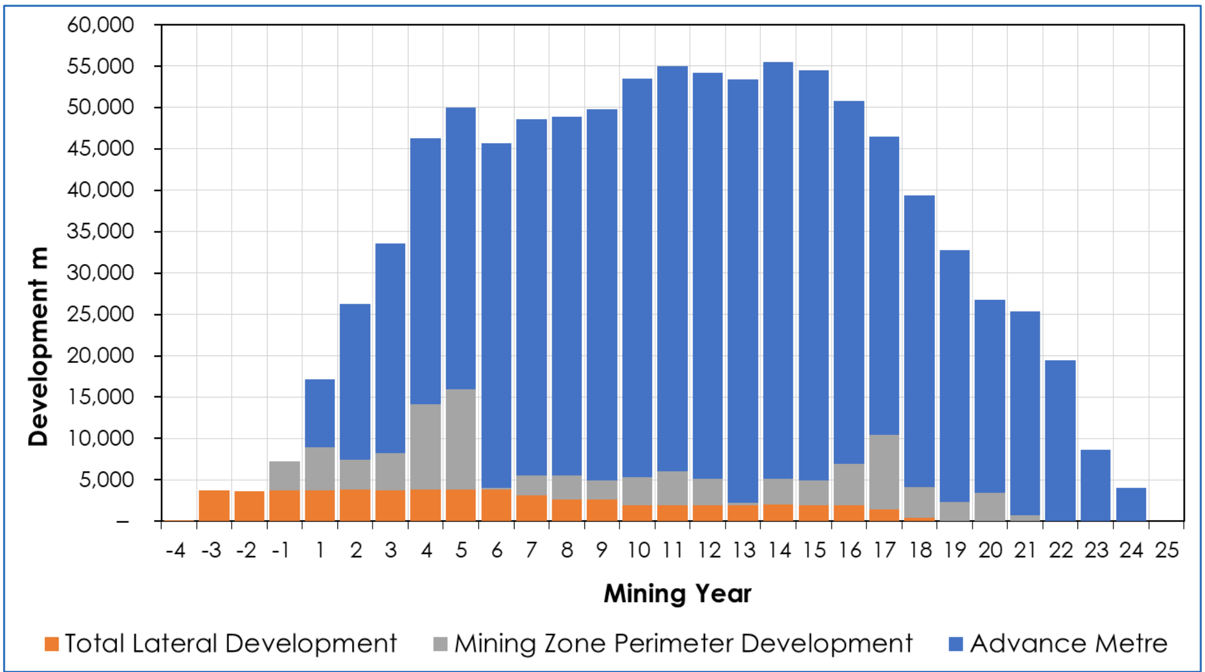


Figure by OreWin, 2017.

Figure 24.18 Kakula 6 Mtpa PEA by Resource Category

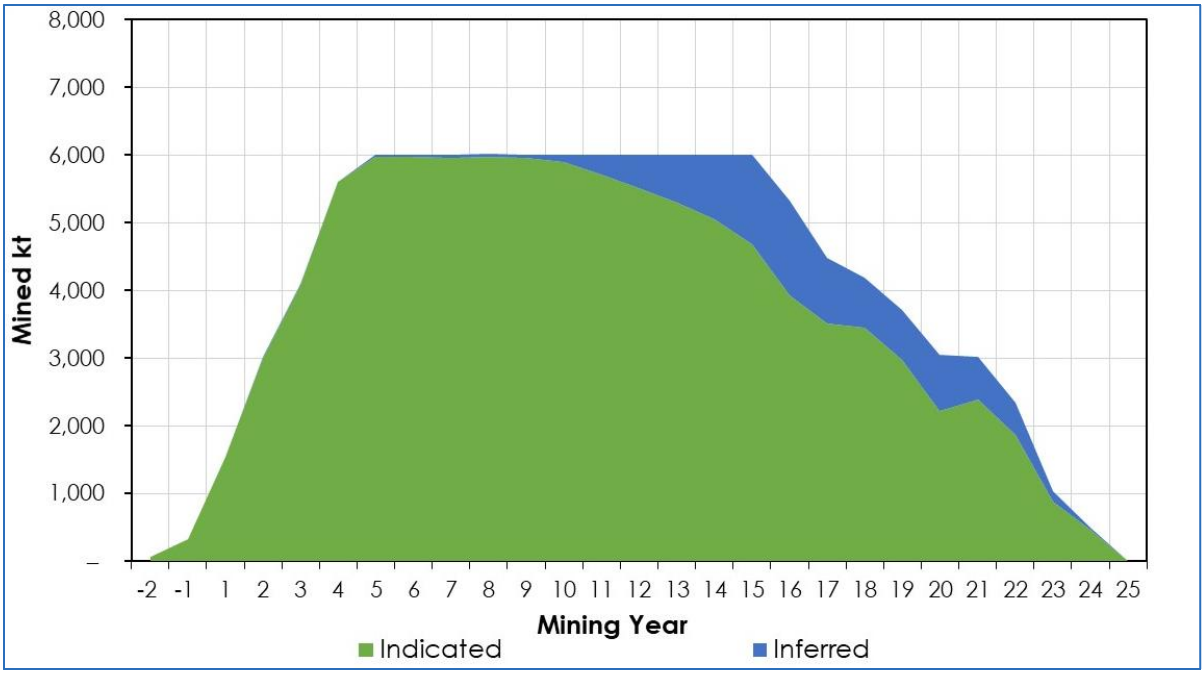


Figure by OreWin, 2017.

Table 24.23 Kakula 6 Mtpa PEA Production

Mining Year	Mined (kt)	Cu (%)	Fe (%)	As (%)	S (%)	NSR17 (US$/t)
-3	–	–	–	–	–	–
-2	73	3.30	4.29	0.00	0.86	146.99
-1	324	5.60	4.69	0.00	1.43	253.26
1	1,549	5.52	4.89	0.00	1.41	249.09
2	3,016	6.54	4.67	0.00	1.66	296.98
3	4,095	7.48	4.77	0.00	1.90	340.75
4	5,611	7.30	4.80	–	1.89	332.36
5	6,007	6.46	4.77	–	1.70	292.74
6	6,010	6.44	4.74	–	1.70	292.12
7	6,013	6.34	4.75	–	1.68	287.14
8	6,018	6.30	4.75	–	1.67	285.42
9	6,012	6.10	4.72	–	1.63	276.29
10	6,002	5.57	4.82	–	1.52	251.50
11	6,009	5.46	4.85	–	1.52	246.13
12	6,004	5.19	4.94	–	1.50	233.53
13	6,007	5.05	4.93	–	1.45	226.85
14	6,009	4.97	4.89	–	1.39	223.17
15	6,000	4.76	4.88	–	1.34	213.36
16	5,326	4.54	4.83	–	1.32	202.96
17	4,482	4.28	4.94	–	1.36	191.10
18	4,184	4.21	4.94	–	1.31	187.87
19	3,718	4.14	4.95	–	1.30	184.67
20	3,048	4.05	5.09	–	1.31	180.78
21	3,020	4.02	5.05	–	1.30	179.37
22	2,350	4.01	4.98	–	1.24	178.75
23	1,045	3.92	4.76	–	1.07	174.57
24	492	3.81	4.64	–	0.91	169.34
Total	108,422	5.48	4.85	0.00	1.52	247.34

24.4.2 Kansoko Mine Schedule

Kansoko mining was scheduled from the Kamoa 2017 PFS design assuming that the mining method was controlled convergence room-and-pillar and the mining rate was 6 Mtpa. Figure 24.19 and Table 24.24 show the 6 Mtpa tonnes and grades mined at the Kansoko Mine. Over the Kansoko mine life, 89% of the overall tonnes are mined using controlled convergence room-and-pillar, 3% are mined using room-and-pillar, with the remaining 8% being from development. Figure 24.20 shows the Kansoko Mine quantity of material mined split by resource category.

Figure 24.19 Kansoko Mine Production

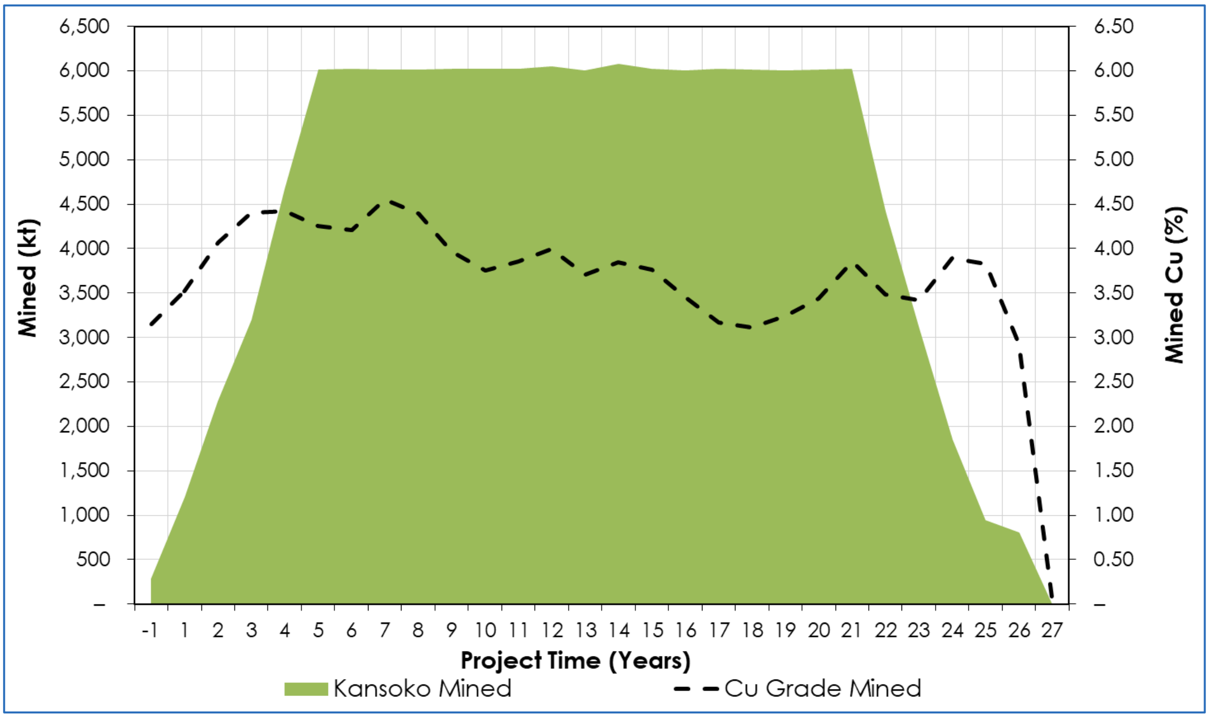


Figure by OreWin, 2017.

Table 24.24 Kamoa 2017 PFS 6 Mtpa Mine Production

Year (no.)	Mined Tonnes (kt)	Cu (%)	AsCu (%)	Fe (%)	As (%)	S (%)	NSR ($/t)
-1	289	3.15	0.23	6.11	0.00	2.35	137.42
1	1,206	3.52	0.29	5.56	0.00	2.12	155.01
2	2,282	4.07	0.32	5.70	0.00	2.34	181.61
3	3,205	4.40	0.35	6.12	0.00	2.76	196.18
4	4,667	4.42	0.29	6.30	0.00	2.71	196.73
5	6,014	4.26	0.32	6.83	0.00	2.95	188.96
6	6,028	4.21	0.29	6.68	0.00	3.14	186.08
7	6,010	4.55	0.26	6.37	0.00	3.14	201.53
8	6,015	4.39	0.29	6.15	0.00	2.86	194.66
9	6,027	3.97	0.34	6.01	0.00	2.28	174.66
10	6,027	3.75	0.25	6.10	0.00	2.44	164.46
11	6,027	3.85	0.29	6.26	0.00	2.67	169.58
12	6,050	3.99	0.34	6.24	0.00	2.64	176.11
13	6,007	3.71	0.31	6.25	0.00	2.45	162.84
14	6,077	3.85	0.38	6.21	0.00	2.48	169.32
15	6,022	3.76	0.40	6.08	0.00	2.41	165.00
16	6,005	3.46	0.45	6.15	0.00	2.34	151.61
17	6,019	3.17	0.33	6.12	0.00	2.41	138.70
18	6,012	3.12	0.29	6.08	0.00	2.24	136.24
19	6,009	3.24	0.29	6.26	0.00	2.39	142.12
20	6,018	3.44	0.27	5.78	0.00	2.09	150.50
21	6,022	3.86	0.32	5.70	0.00	2.11	169.98
22	4,417	3.49	0.29	5.78	0.00	2.18	153.42
23	3,126	3.42	0.32	5.85	0.00	2.24	149.60
24	1,858	3.89	0.50	5.72	0.00	1.76	172.36
25	943	3.83	0.75	6.03	0.00	1.34	171.20
26	802	2.93	0.51	5.59	0.00	1.30	128.72
Total	125,182	3.81	0.32	6.14	0.00	2.49	168.06

Figure 24.20 Kamoa 2017 PFS Mine Production by Resource Category

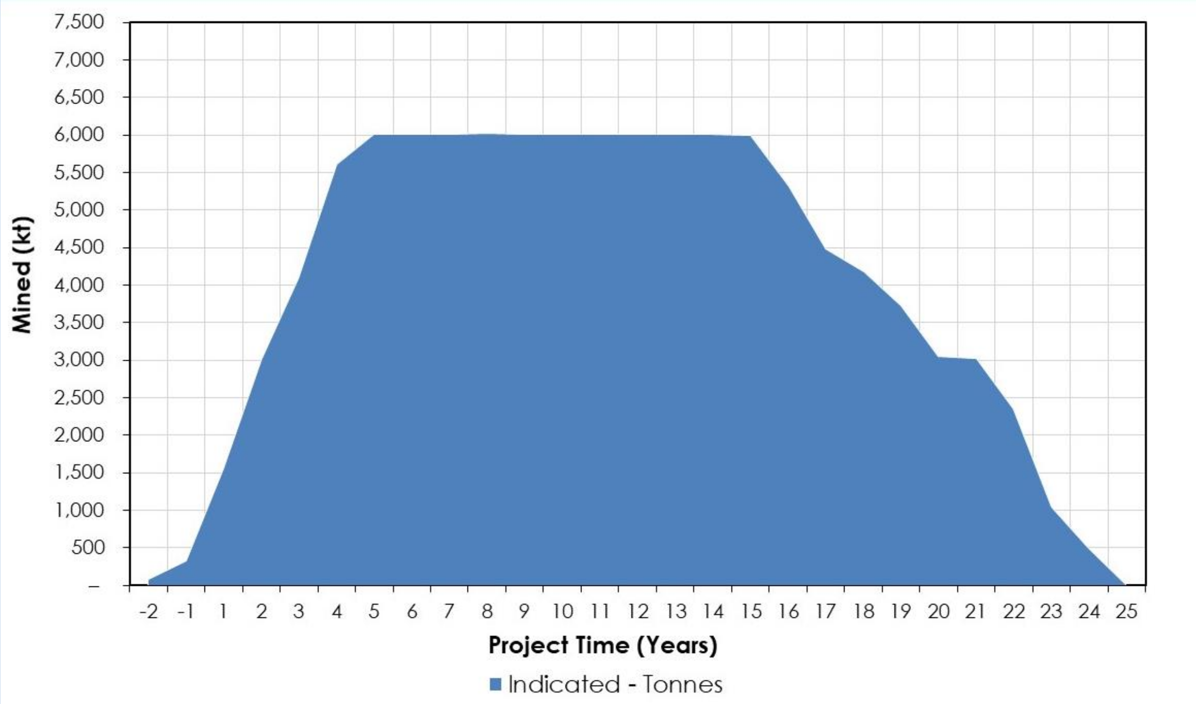


Figure by OreWin, 2017.

24.4.3 Kamoa-Kakula 12 Mtpa PEA Mine Schedule

The Kamoa-Kakula 12 Mtpa PEA assesses the development of both the Kakula and Kamoa deposits as an integrated, 12 Mtpa mining and processing complex. Each operation is expected to be a separate underground mine with associated dedicated processing facilities and surface infrastructure.

The Kamoa-Kakula 12 Mtpa PEA scenario envisages the construction and operation of two separate facilities: the Kakula 6 Mtpa PEA on the Kakula Deposit and the Kansoko Mine on the Kansoko Sud and Kansoko Centrale areas of the Kamoa Deposit. The Kakula 6 Mtpa PEA scenario is the same as that presented in the Kakula 2017 PEA 6 Mtpa. The initial plant capacity of 3 Mtpa is expanded to 6 Mtpa as the Kansoko Mine and Kakula 6 Mtpa PEA ramp up. The mines continue to ramp up to 12 Mtpa combined by Year Nine. Once the Kansoko and Kakula 6 Mtpa PEAs near the end of their mine life, Kamoa North comes on line to maintain the overall production at 12 Mtpa. The 12 Mtpa PEA also analyses an on-site smelter to produce blister copper at the mine site.

Kamoa-Kakula 12 Mtpa PEA mining production is shown in Figure 24.16 and Table 24.23.

Figure 24.21 Kamoa-Kakula 12 Mtpa PEA Mine Production

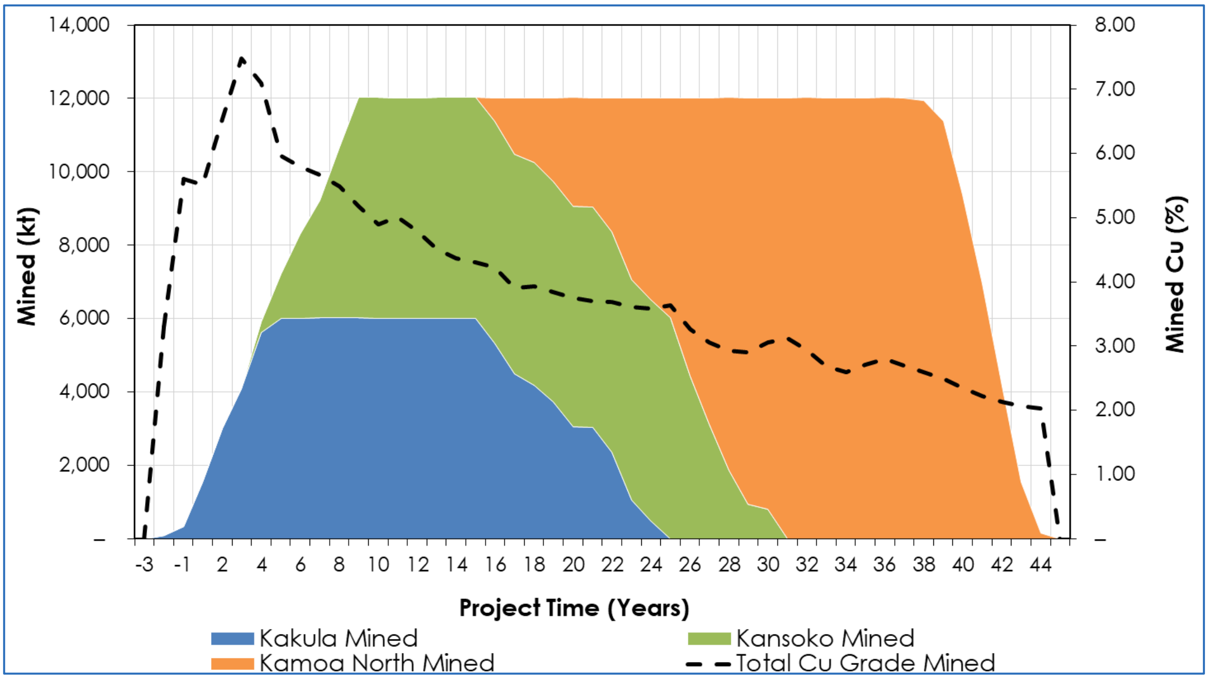


Figure by OreWin, 2017.

Table 24.25 Kamoa-Kakula 12 Mtpa PEA Mine Production

Mining Year	Mined (kt)	Cu (%)	Fe (%)	As (%)	S (%)	Mining Year	Mined (kt)	Cu (%)	Fe (%)	As (%)	S (%)
-3	–	–	–	–	–	25	12,007	3.64	5.65	0.00	2.07
-2	73	3.30	4.29	0.00	0.86	26	12,007	3.26	5.67	0.00	2.05
-1	324	5.60	4.69	0.00	1.43	27	12,013	3.05	5.69	0.00	1.98
1	1,549	5.52	4.89	0.00	1.41	28	12,034	2.93	5.66	0.00	1.82
2	3,016	6.54	4.67	0.00	1.66	29	12,001	2.90	5.56	0.00	1.70
3	4,095	7.48	4.77	0.00	1.90	30	12,007	3.06	5.43	0.00	1.69
4	5,900	7.10	4.87	0.00	1.91	31	12,003	3.12	5.38	0.00	1.68
5	7,213	5.97	4.90	0.00	1.77	32	12,037	2.95	5.34	0.00	1.56
6	8,291	5.79	5.00	0.00	1.88	33	12,004	2.68	5.33	0.00	1.40
7	9,218	5.67	5.23	0.00	2.06	34	12,006	2.59	5.38	0.00	1.40
8	10,685	5.48	5.43	0.00	2.13	35	12,003	2.71	5.48	0.00	1.61
9	12,026	5.18	5.77	0.00	2.29	36	12,037	2.79	5.60	0.00	1.83
10	12,030	4.89	5.75	0.00	2.33	37	12,003	2.69	5.63	0.00	1.83
11	12,018	5.00	5.61	0.00	2.33	38	11,951	2.59	5.63	0.00	1.83
12	12,018	4.79	5.55	0.00	2.18	39	11,384	2.49	5.58	0.00	1.81
13	12,034	4.51	5.47	0.00	1.87	40	9,381	2.35	5.54	0.00	1.75
14	12,036	4.36	5.50	0.00	1.91	41	6,949	2.22	5.51	0.00	1.60
15	12,028	4.31	5.57	0.00	2.01	42	4,147	2.13	5.56	0.00	1.53
16	12,014	4.23	5.67	0.00	2.14	43	1,532	2.07	5.59	0.00	1.50
17	11,998	3.90	5.87	0.00	2.23	44	142	2.03	5.77	0.00	1.50
18	12,003	3.93	5.84	0.00	2.21	45	–	–	–	–	–
19	12,007	3.84	5.77	0.00	2.14	46	–	–	–	–	–
20	12,028	3.75	5.90	0.00	2.13	47	–	–	–	–	–
21	12,007	3.69	5.82	0.00	2.14	48	–	–	–	–	–
22	12,009	3.68	5.78	0.00	2.05	49	–	–	–	–	–
23	12,000	3.61	5.90	0.00	2.17	50	–	–	–	–	–
24	12,010	3.58	5.66	0.00	2.04	Total	444,276	3.79	5.56	0.00	1.93

24.4.4 Kamoa-Kakula Process Production

Table 24.26 shows the concentrator assumptions used for the calculation of the copper recovery, net smelter return and revenue for Kamoa-Kakula 2017 PFS and PEA scenarios.

Table 24.26 Plant Recovery and Concentrate Revenue Factors

Description	Unit	Kakula	Kansoko		Kamoa North	
			Supergene	Hypogene	Supergene	Hypogene
Cu Feed Reference	%	6.01	3.54	3.54	3.54	3.54
Concentrator Assumptions						
Cu Tail Reference	%	0.86	0.43	0.44	0.43	0.44
Cu nf Reference	%	0.10	0.15	0.15	0.15	0.15
Cu Recovery Reference	%	87.09	88.67	88.67	88.67	88.67
Concentrate Cu Grade	%	54.94	45.00	35.97	45.00	35.97
Concentrator S Grade	%	14.60	16.90	31.60	16.90	31.60
CU:S	%	3.76	2.67	1.14	2.67	1.14
Concentrator Moisture	%	12.00	12.00	12.00	12.00	12.00
Concentrate Payability	%	97.63	97.11	96.39	97.11	96.39

Plant feed, concentrate and copper production for the Kakula 2017 PEA 6 Mtpa are summarised in Figure 24.22 and Figure 24.23. Plant feed, concentrate and copper production for the Kamoa 2017 PFS 6 Mtpa are summarised in Figure 24.24 and Figure 24.25 and plant feed, concentrate and copper production for the Kamoa-Kakula 12 Mtpa PEA are summarised in Figure 24.26 and Figure 24.27.

Figure 24.22 Plant Feed and Copper Head Grade for Kakula 2017 PEA 6 Mtpa

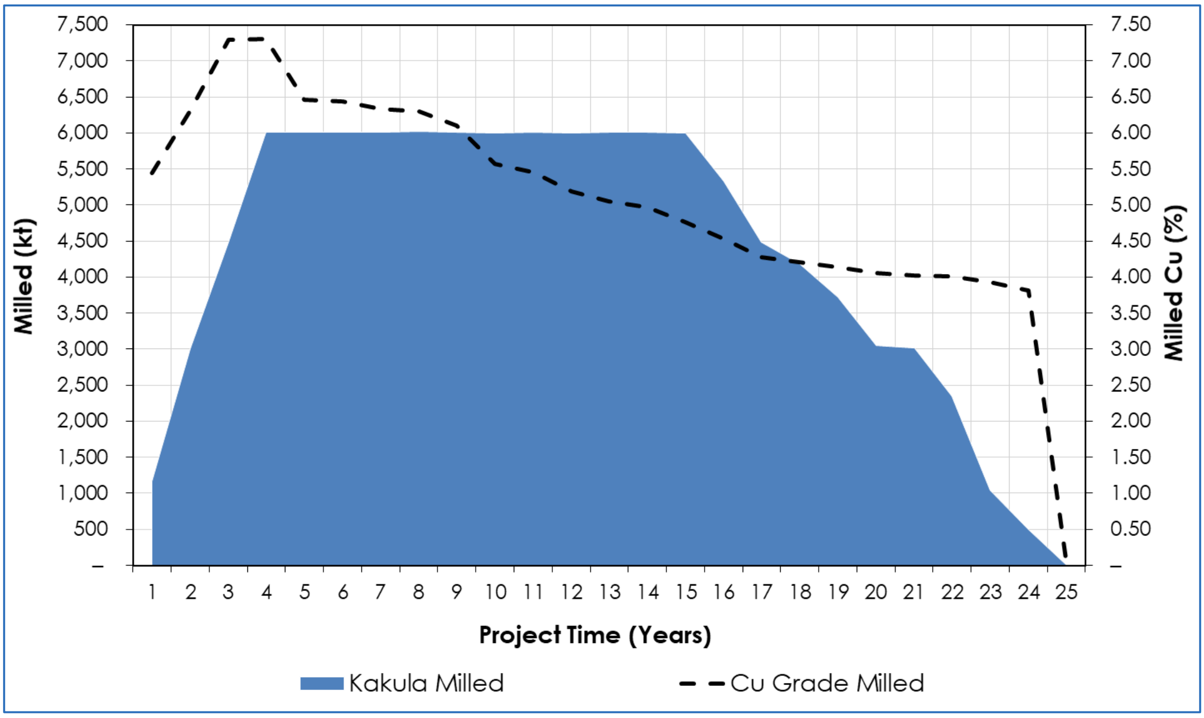


Figure by OreWin, 2017.

Figure 24.23 Concentrate Produced for Kakula 2017 PEA 6 Mtpa

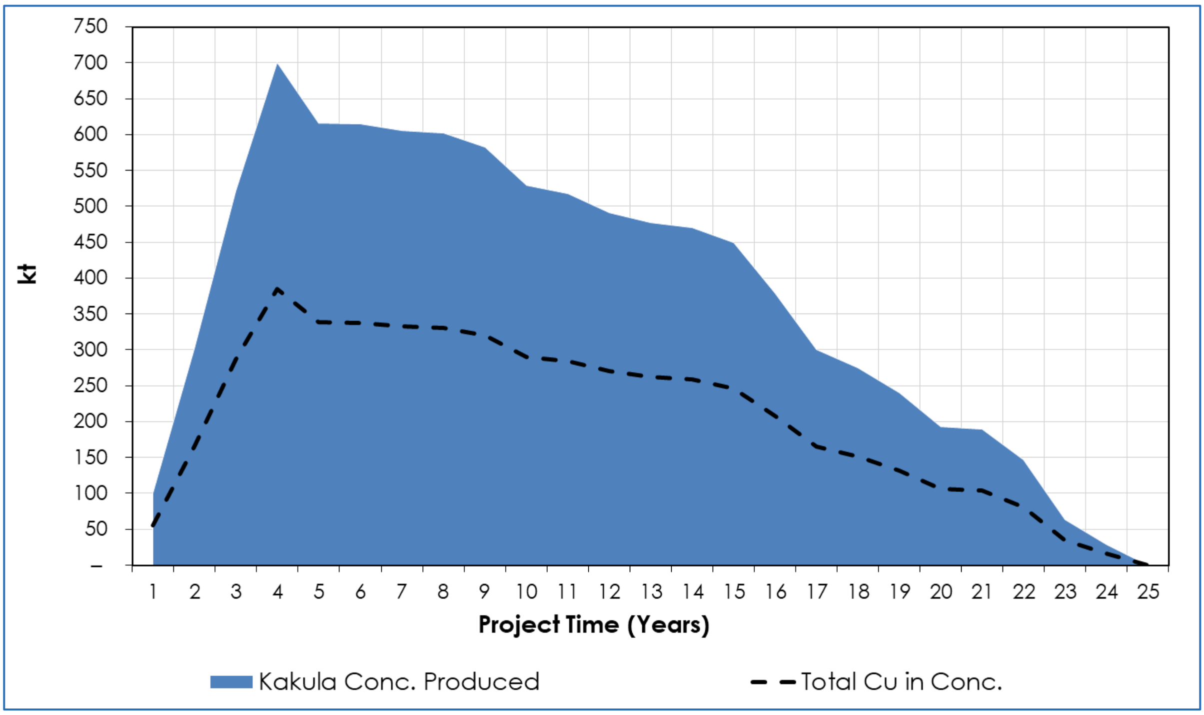


Figure by OreWin, 2017.

Figure 24.24 Plant Feed and Copper Head Grade for Kamoa 2017 PFS 6 Mtpa

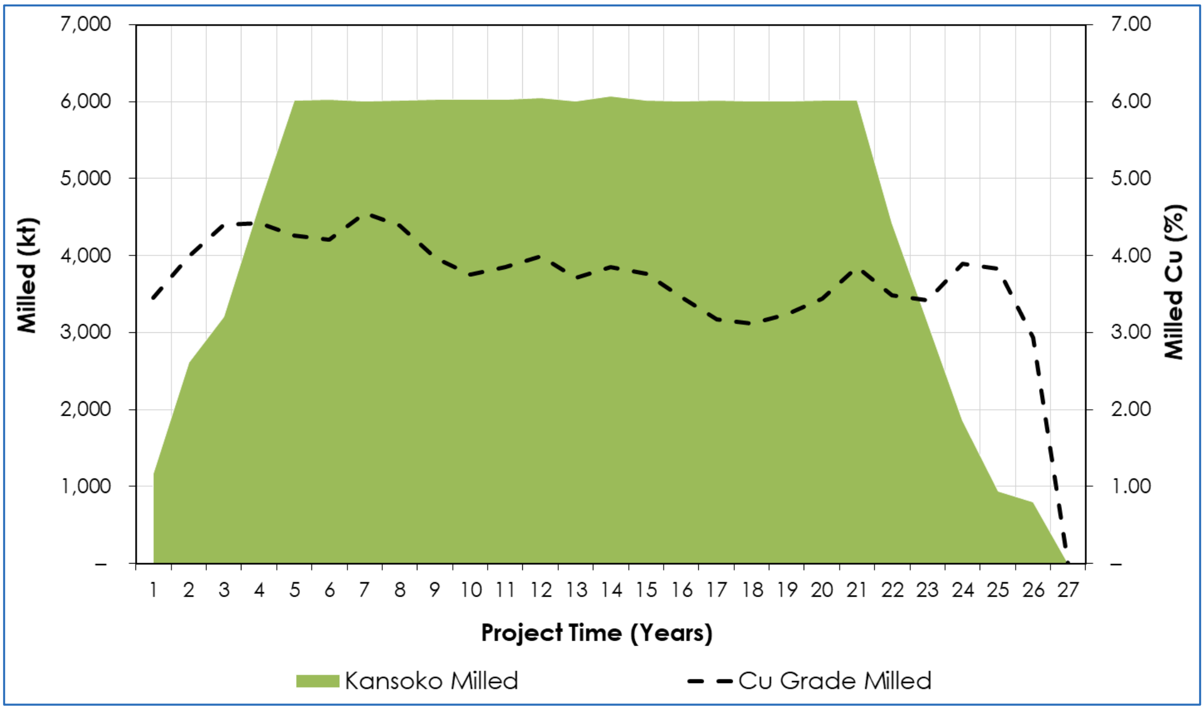


Figure by OreWin, 2017.

Figure 24.25 Concentrate Produced for Kamoa 2017 PFS 6 Mtpa

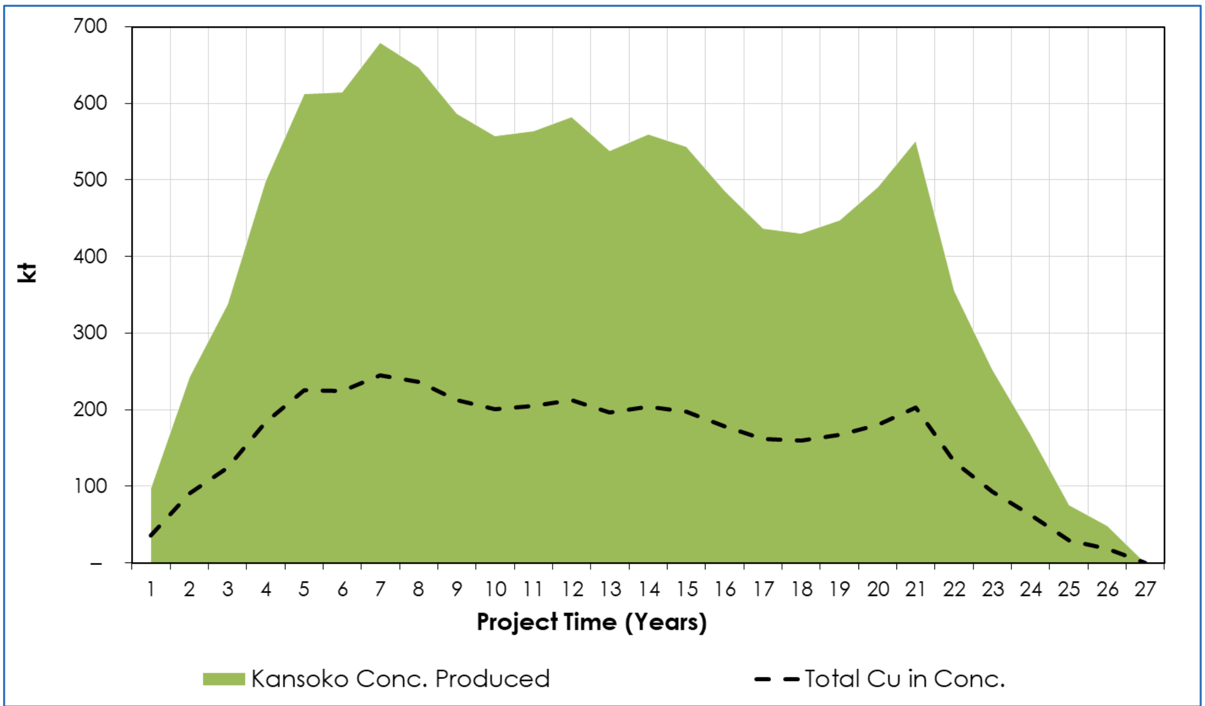


Figure by OreWin, 2017.

Figure 24.26 Plant Feed and Copper Head Grade for Kamoa-Kakula 12 Mtpa PEA

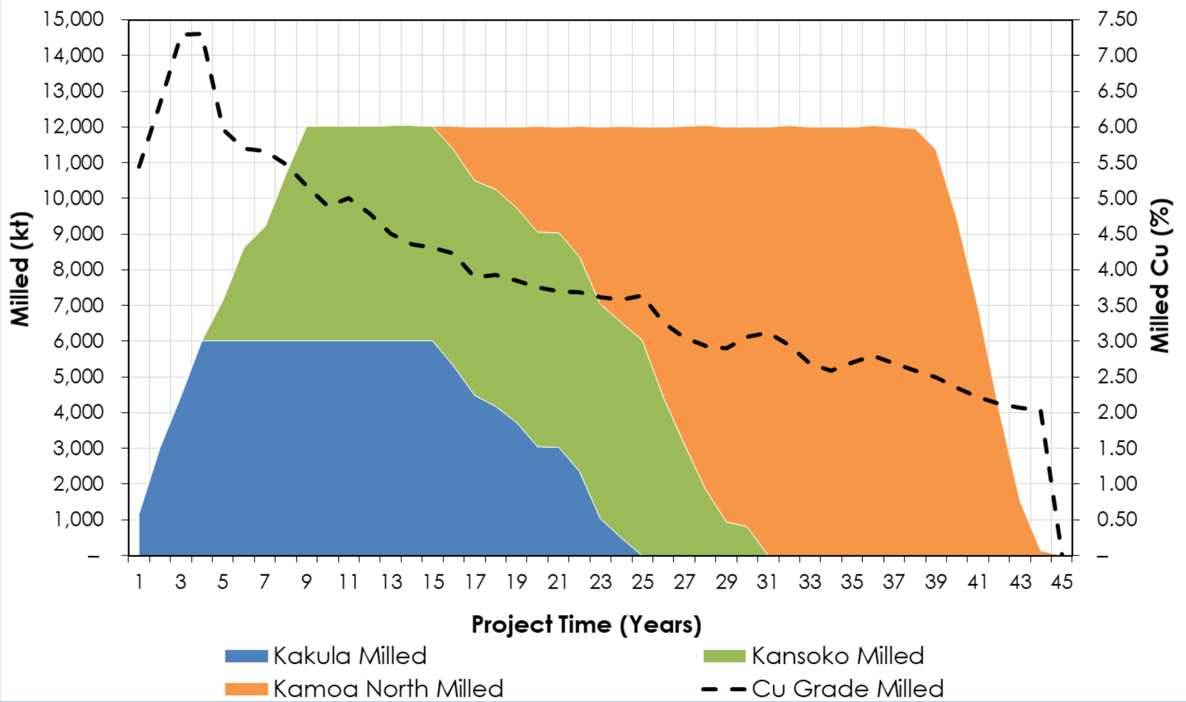


Figure by OreWin, 2017.

Figure 24.27 Concentrate Produced for Kamoa-Kakula 12 Mtpa PEA

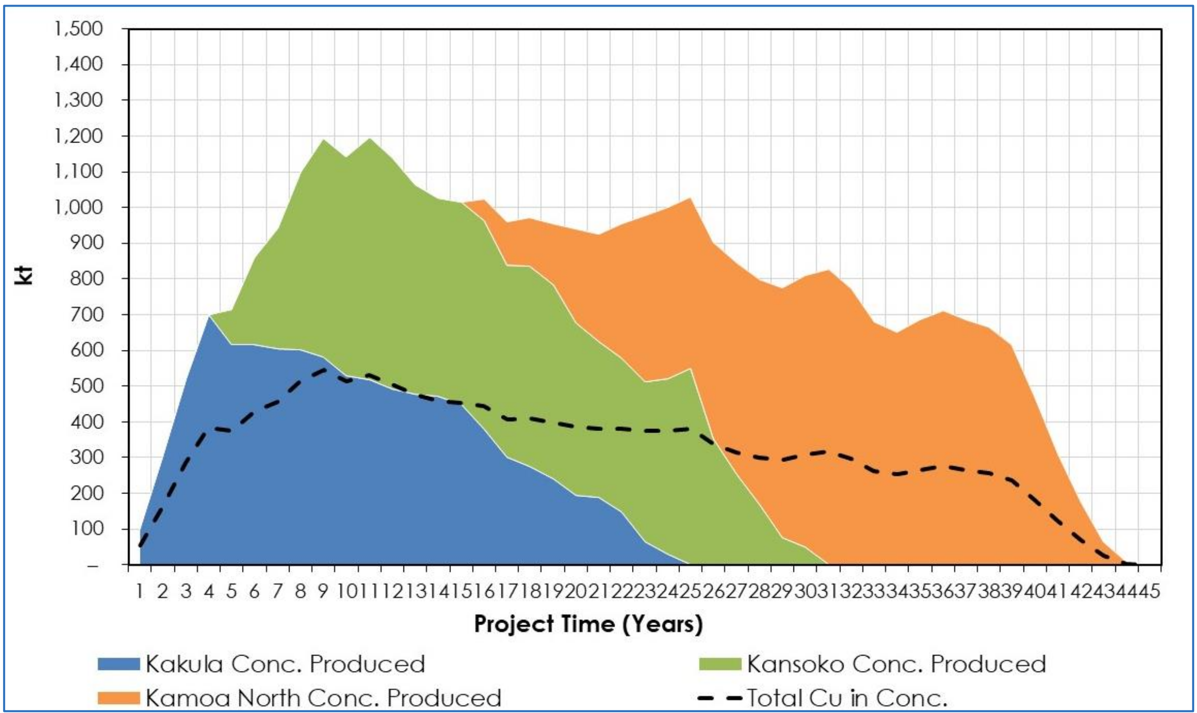


Figure by OreWin, 2017.

Table 24.27 Kakula 2017 PEA 6 Mtpa Processing Production Schedule

Description	Units	Totals	Project Time (Years)														
			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Quantity Milled	kt	108,422	1,169	3,006	4,484	6,010	6,007	6,010	6,013	6,018	6,012	6,002	6,009	6,004	6,007	6,009	6,000
Cu Feed Grade	% Cu	5.48	5.45	6.31	7.29	7.30	6.46	6.44	6.34	6.30	6.10	5.57	5.46	5.19	5.05	4.97	4.76
Copper Conc. Produced	kt (dry)	9,400	101	301	521	700	616	615	605	602	582	529	518	491	478	470	449
Copper Concentrate Recovery	%	86.86	86.80	87.23	87.65	87.65	87.30	87.29	87.24	87.23	87.14	86.87	86.81	86.66	86.57	86.53	86.39
Copper Concentrate Grade	% Cu	54.94	54.94	54.94	54.94	54.94	54.94	54.94	54.94	54.94	54.94	54.94	54.94	54.94	54.94	54.94	54.94
Total Recovered Copper Production	Mlb	11,385	122	365	632	848	746	745	733	729	705	641	628	595	579	570	544
Total Recovered Copper Production	kt	5,164	55	166	286	385	339	338	332	331	320	291	285	270	263	258	247
Description	**Units**	**Project Time (Years)**															
		16	**17**	**18**	**19**	**20**	**21**	**22**	**23**	**24**	**25**	**26**	**27**	**28**	**29**	**30**	**31**
Quantity Milled	kt	5,326	4,482	4,184	3,718	3,048	3,020	2,350	1,045	492	–	–	–	–	–	–	–
Cu Feed Grade	% Cu	4.54	4.28	4.21	4.14	4.05	4.02	4.01	3.92	3.81	–	–	–	–	–	–	–
Copper Conc. Produced	kt (dry)	379	300	276	241	193	190	147	64	29	–	–	–	–	–	–	–
Copper Concentrate Recovery	%	86.24	86.06	86.00	85.95	85.88	85.86	85.85	85.77	85.68	–	–	–	–	–	–	–
Copper Concentrate Grade	% Cu	54.94	54.94	54.94	54.94	54.94	54.94	54.94	54.94	54.94	–	–	–	–	–	–	–
Total Recovered Copper Production	Mlb	459	364	334	292	234	230	178	78	35	–	–	–	–	–	–	–
Total Recovered Copper Production	kt	208	165	151	132	106	104	81	35	16	–	–	–	–	–	–	–

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Table 24.28 Kamoa 2017 PFS 6 Mtpa Processing Production Schedule

Description	Units	Totals	Project Time (Years)														
			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Quantity Milled	kt	125,182	1,169	2,608	3,205	4,667	6,014	6,028	6,010	6,015	6,027	6,027	6,027	6,050	6,007	6,077	6,022
Cu Feed Grade	% Cu	3.81	3.45	3.99	4.40	4.42	4.26	4.21	4.55	4.39	3.97	3.75	3.85	3.99	3.71	3.85	3.76
Copper Conc. Produced	kt (dry)	11,405	97	243	338	498	613	614	679	648	587	558	564	583	539	560	544
Copper Concentrate Recovery	%	87.52	87.77	87.05	88.11	88.64	88.17	88.55	89.72	89.31	88.72	88.82	88.34	88.03	87.87	87.13	87.22
Copper Concentrate Grade	% Cu	36.63	36.35	37.35	36.81	36.76	36.84	36.54	36.12	36.43	36.16	36.04	36.36	36.48	36.36	36.39	36.32
Total Recovered Copper Production	Mlb	9,211	78	200	274	403	498	495	541	520	468	443	452	469	432	449	435
Total Recovered Copper Production	kt	4,178	35	91	124	183	226	224	245	236	212	201	205	213	196	204	198
Description	Units	Project Time (Years)															
		16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31
Quantity Milled	kt	6,005	6,019	6,012	6,009	6,018	6,022	4,417	3,126	1,858	943	802	–	–	–	–	–
Cu Feed Grade	% Cu	3.46	3.17	3.12	3.24	3.44	3.86	3.49	3.42	3.89	3.83	2.93	–	–	–	–	–
Copper Conc. Produced	kt (dry)	485	436	431	448	490	551	355	253	168	76	49	–	–	–	–	–
Copper Concentrate Recovery	%	85.69	84.60	85.47	86.03	87.20	87.29	85.47	87.06	86.65	79.86	79.83	–	–	–	–	–
Copper Concentrate Grade	% Cu	36.69	37.01	37.20	37.44	36.79	36.78	37.13	36.74	37.26	38.15	38.29	–	–	–	–	–
Total Recovered Copper Production	Mlb	392	356	353	369	398	447	290	205	138	64	41	–	–	–	–	–
Total Recovered Copper Production	kt	178	162	160	168	180	203	132	93	63	29	19	–	–	–	–	–

Table 24.29 Kamoa – Kakula 2017 PEA 12 Mtpa Processing Production Schedule

Description	Units	Totals	Project Time (Years)														
			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Quantity Milled	kt	444,276	1,169	3,006	4,484	6,010	7,176	8,618	9,218	10,685	12,026	12,030	12,018	12,018	12,034	12,036	12,028
Cu Feed Grade	% Cu	3.79	5.45	6.31	7.29	7.30	5.97	5.70	5.67	5.48	5.18	4.89	5.00	4.79	4.51	4.36	4.31
Copper Conc. Produced	kt (dry)	34,206	101	301	521	700	714	858	943	1,100	1,195	1,143	1,197	1,139	1,065	1,028	1,014
Copper Conc. - External Smelter	kt (dry)	9,744	101	301	521	700	24	168	253	410	505	453	507	449	375	338	324
Copper Conc. -Internal Smelter	kt (dry)	24,461	–	–	–	–	690	690	690	690	690	690	690	690	690	690	690
Copper Concentrate Recovery	%	85.97	86.80	87.23	87.65	87.65	87.34	87.24	87.48	87.72	87.56	87.60	88.13	87.88	87.52	87.51	87.27
Copper Concentrate Grade	% Cu	42.30	54.94	54.94	54.94	54.94	52.40	49.97	48.44	46.71	45.66	45.05	44.27	44.42	44.59	44.69	44.59
Contained Copper in Conc. - External Smelter	Mlb	10,627	122	365	632	848	29	203	306	496	608	539	571	519	449	409	392
Contained Copper in Conc. - External Smelter	kt	4,820	55	166	286	385	13	92	139	225	276	245	259	235	204	186	178
Contained Copper in Blister - Internal Smelter	Mlb	20,955	–	–	–	–	784	731	690	627	586	587	588	588	588	594	595
Contained Copper in Blister - Internal Smelter	kt	9,505	–	–	–	–	356	331	313	284	266	266	267	267	267	270	270
Total Recovered Copper Production	Mlb	31,582	122	365	632	848	813	934	997	1,123	1,194	1,127	1,160	1,106	1,038	1,004	987
Total Recovered Copper Production	kt	14,325	55	166	286	385	369	424	452	509	542	511	526	502	471	455	448

Description	Units	Project Time (Years)															
		16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31
Quantity Milled	kt	12,014	11,998	12,003	12,007	12,028	12,007	12,009	12,000	12,010	12,007	12,007	12,013	12,034	12,001	12,007	12,003
Cu Feed Grade	% Cu	4.23	3.90	3.93	3.84	3.75	3.69	3.68	3.61	3.58	3.64	3.26	3.05	2.93	2.90	3.06	3.12
Copper Conc. Produced	kt (dry)	1,023	959	971	953	939	926	954	979	1,002	1,031	903	843	800	774	811	828
Copper Conc. - External Smelter	kt (dry)	333	269	281	263	249	236	264	289	312	341	213	153	110	84	121	138
Copper Conc. -Internal Smelter	kt (dry)	690	690	690	690	690	690	690	690	690	690	690	690	690	690	690	690
Copper Concentrate Recovery	%	87.20	87.02	86.57	86.36	85.67	85.59	86.29	86.74	87.20	87.10	85.79	85.80	85.07	84.08	84.35	84.37
Copper Concentrate Grade	% Cu	43.36	42.44	42.07	41.82	41.18	41.00	40.00	38.41	37.46	36.92	37.26	37.28	37.52	37.86	38.18	38.21
Contained Copper in Conc. - External Smelter	Mlb	381	302	306	290	273	253	273	262	265	277	175	126	91	70	102	116
Contained Copper in Conc. - External Smelter	kt	173	137	139	132	124	115	124	119	120	126	79	57	41	32	46	53
Contained Copper in Blister - Internal Smelter	Mlb	587	587	585	580	571	575	560	558	553	553	558	559	562	567	572	573
Contained Copper in Blister - Internal Smelter	kt	266	266	265	263	259	261	254	253	251	251	253	253	255	257	259	260
Total Recovered Copper Production	Mlb	968	889	891	870	844	828	833	820	819	831	733	685	653	638	674	688
Total Recovered Copper Production	kt	439	403	404	395	383	376	378	372	371	377	332	311	296	289	306	312

Description	Units	Project Time (Years)															
		32	33	34	35	36	37	38	39	40	41	42	43	44	45	46	47
Quantity Milled	kt	12,037	12,004	12,006	12,003	12,037	12,003	11,951	11,384	9,381	6,949	4,147	1,532	142	–	–	–
Cu Feed Grade	% Cu	2.95	2.68	2.59	2.71	2.79	2.69	2.59	2.49	2.35	2.22	2.13	2.07	2.03	–	–	–
Copper Conc. Produced	kt (dry)	771	681	651	685	712	686	666	617	466	311	176	65	6	–	–	–
Copper Conc. - External Smelter	kt (dry)	81	–	–	–	22	–	–	–	–	311	176	65	6	–	–	–
Copper Conc. -Internal Smelter	kt (dry)	690	681	651	685	690	686	666	617	466	–	–	–	–	–	–	–
Copper Concentrate Recovery	%	83.41	81.89	81.31	81.89	82.33	82.44	82.85	83.16	81.96	80.12	79.52	80.14	81.43	–	–	–
Copper Concentrate Grade	% Cu	38.33	38.65	38.81	38.81	38.90	38.86	38.56	38.26	38.75	39.68	39.87	39.22	37.80	–	–	–
Contained Copper in Conc. - External Smelter	Mlb	69	–	–	–	19	–	–	–	–	272	155	56	5	–	–	–
Contained Copper in Conc. - External Smelter	kt	31	–	–	–	8	–	–	–	–	124	70	25	2	–	–	–
Contained Copper in Blister - Internal Smelter	Mlb	574	572	548	577	583	579	557	512	392	–	–	–	–	–	–	–
Contained Copper in Blister - Internal Smelter	kt	261	259	249	262	264	263	253	232	178	–	–	–	–	–	–	–
Total Recovered Copper Production	Mlb	643	572	548	577	601	579	557	512	392	272	155	56	5	–	–	–
Total Recovered Copper Production	kt	292	259	249	262	273	263	253	232	178	124	70	25	2	–	–	–

24.5 Kakula 2017 PEA Mining

24.5.1 Stope Optimisation and Mining Cut-off Grades

Stope optimisation was undertaken on the resource model at mining cut-off grades of 6.6% Cu and 3.0% Cu. A dilution allowance of 30 cm on footwall and hangingwall was added to the model. The resulting stope shapes were then further optimised by height. Figure 24.28 shows the optimised block categorised into areas less than 6 m and areas more than 6 m.

Figure 24.28 Stope Optimisation Blocks by Height

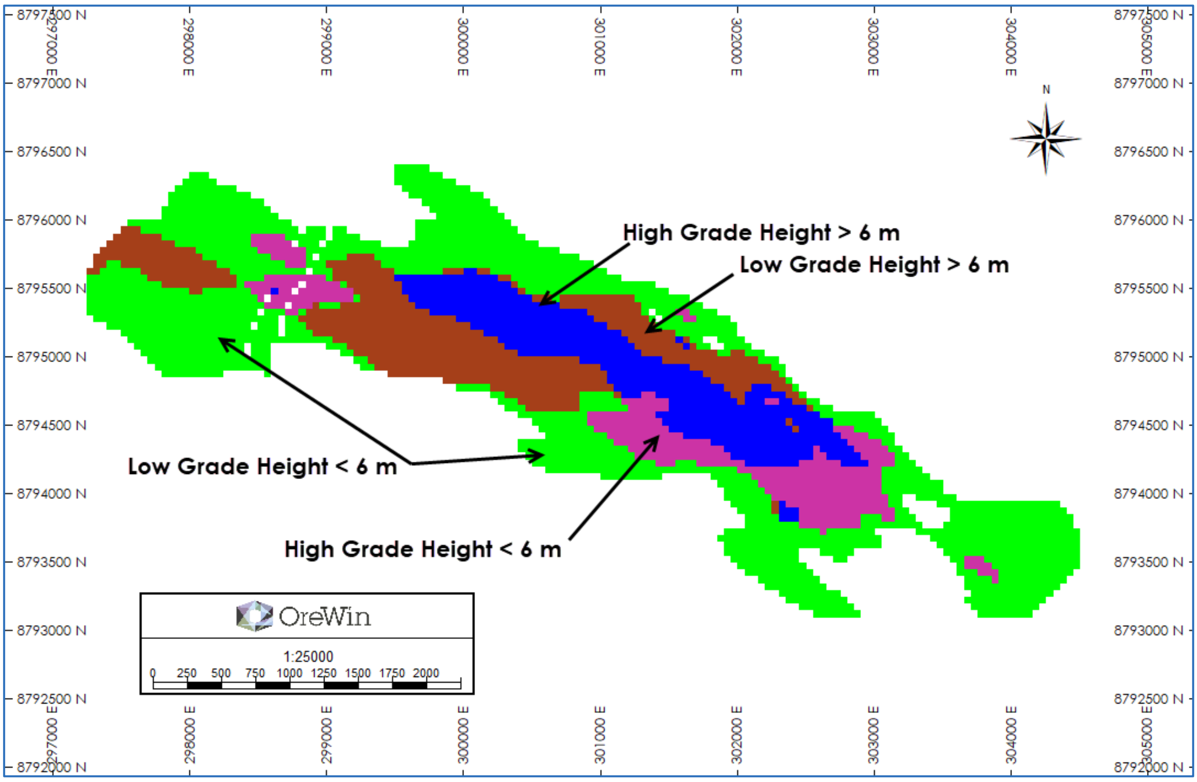

Figure by OreWin, 2017.

24.5.2 Mining Methods

Two mining methods were chosen:

- Controlled convergence room-and-pillar.
- Drift-and-fill with pastefill.

The controlled convergence room-and-pillar method was selected for heights greater than 3 m and less than 6 m and dip less than 25 degrees. The drift-and-fill with pastefill was selected for heights greater than 6 m. The drift-and-fill with pastefill method was also selected for heights greater than 3 m and less than 6 m and dip greater than 25 degrees. The maximum height from the optimisation was 15.1 m.

Two drift-and-fill mining lifts were selected for the stope heights greater than 6.00 m. The maximum height of the drift-and-fill mining panel is 6.00 m and the minimum height of the drift-and-fill mining panel is 3.00 m.

24.5.2.1 Controlled Convergence Room-and-Pillar

The controlled convergence room-and-pillar method (3 m to 6 m high) allows in-panel pillars to be stripped so the backs and floors can converge in a controlled manner meaning no backfill is required. The protection pillars between the mine workings and preparatory workings are successively extracted as the mining front progresses. Typical extraction ratios are shown in Table 24.30 and Figure 24.29.

Table 24.30 Controlled Convergence Room-and-Pillar in Panel Extraction

Deposit Dip	Thickness (mining height)	Extraction ratio for mining panel
up to 12 Degrees	3–4 m	92.00%
	4–6 m	90.00%
up to 16 Degrees	3–4 m	90.00%
	4–6 m	86.00%
up to 25 Degrees	3–4 m	86.00%
	4–6 m	80.00%

Figure 24.29 Controlled Convergence Room-and-Pillar in Panel Extraction

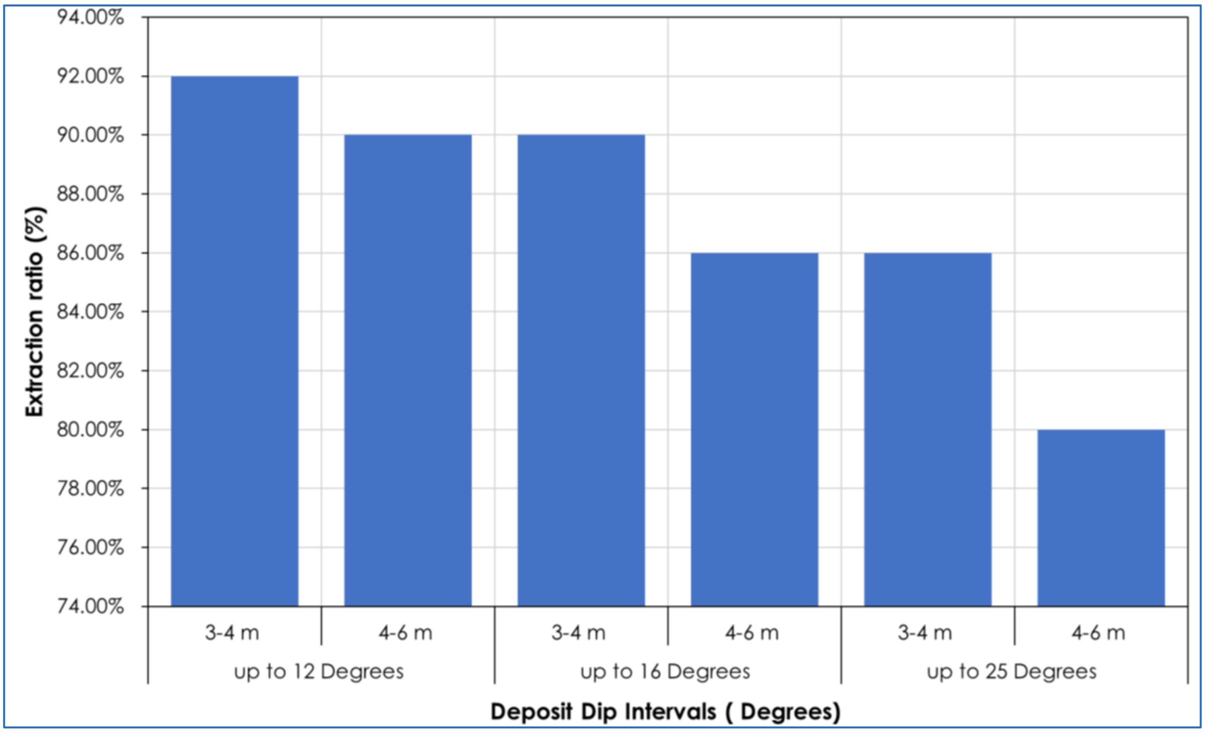

Figure by OreWin, 2017.

The extraction of material using the controlled convergence room-and-pillar method is dependent on the dip and height of the panel. A general design layout for panels dipping from 0° up to 12° is shown in Figure 24.30.

Figure 24.30 Controlled Convergence Room-and-Pillar Panel Design (0° to 12° Dip)

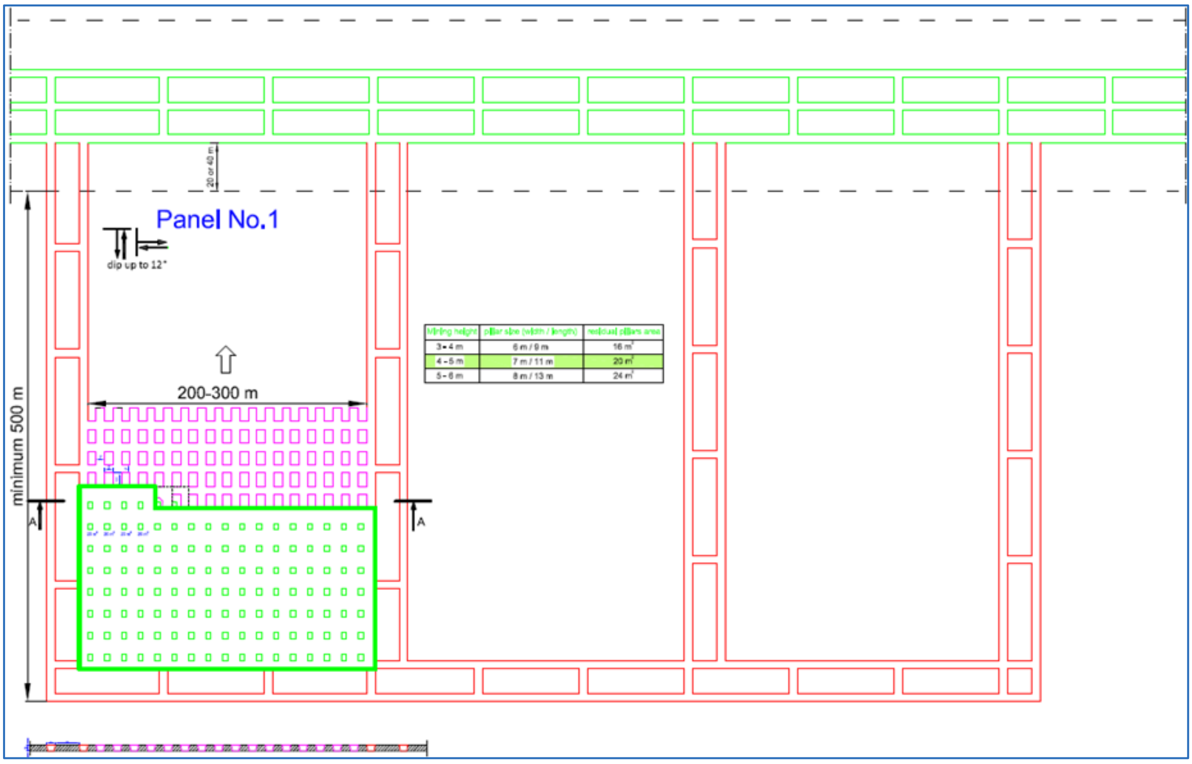

Figure by KGHM Cuprum, 2016.

24.5.2.2 Drift-and-Fill with Pastefill

Drift-and-Fill mining method is a selective underground mining method and ideal for steeply dipping high-grade deposits. The drift-and-fill mining panels would be mined in a primary, secondary, and tertiary sequence and an extraction of 95% was calculated for the drift-and-fill panels.

24.5.3 Mine Design

Selection of the mining method was dictated by mining height and dip. Mine panels were spilt by grade into primary and secondary stages for scheduling. Primary panels contain Cu>6.6% and secondary panels contain Cu<6.6% and Cu>3%.

The decline was designed to accommodate two parallel drives dipping at -18%, one for personnel and machinery, the other for a conveyor. Material mined will be hauled to the transfer points and then transported by conveyor to surface.

The conveyor decline measures 7.0 m (W) x 6.0 m (H) and the service decline measures 5.5 m (W) x 6.0 m (H). The conveyor and service declines are spaced 13.25 m apart. Every 80 m down decline, a 13.25 m cross-cut between the declines and twin remuck cubbies are required. The conveyor drive dips at -18% run NW SE with dimensions 7.0 m (W) x 6.0 m (H). Personnel and machinery access measures 5.5 m (W) x 6.0 m (H) and does not exceed a dip of -8.5°.

Mine ventilation is achieved through ten upcast 5.0 m (D) ventilation raises in varying intake and exhaust combinations depending on the location of mining and air movement requirement.

Backfill boreholes were strategically designed and located to supply backfill to the drift-and-fill panels over the life of the project.

The following is a list of the key development criteria and assumptions for the Kakula 2017 PEA design:

- Conveyor decline and drives dimension 7.00 m (W) x 6.0 m (H).
- Service decline dimension 5.50 m (W) x 6.0 m (H)A.
- The pillar between twin declines is 13.25 m in width.
- Declines dip at maximum gradient of -18.0 % equal to -10.2°.
- Remuck cubbies 15 m in length every 80 m on sidewall of declines.
- Cross-cut 13.25 m in length every 80 m between declines.
- All other lateral development dimensions 5.50 m (W) x 6.0 m (H).
- Service access at maximum gradient of 8.5°.
- Access Stockpile in waste 15 m in length every 160 m on sidewall of service access.
- Access Stockpile in mining zone 15 m in length every 80 m on sidewall of mining zone access.
- Ventilation diameters of 5.0 m depending on requirements.
- Ore pass diameters of 3.0 m depending on requirements.
- Mining Zone pillars width of 20.0 m. Pillar width between two mining zones is 40.0 m.
- Perimeter drives width of 5.0 m inside mining zone pillars.

The assumptions for Kakula development are shown in Table 24.6.

Table 24.31 Kakula 2017 PEA 6 Mtpa Mine Development Assumptions

Description	Height (m)	Width (m)	Other
Conveyor Decline	6.00	7.00	Maximum gradient of -18.0 % (-10.2°).
Service Decline	6.00	5.50	Maximum gradient of -18.0 % (-10.2°).
Conveyor Drift	6.00	7.00	Maximum gradient of -18.0 % (-10.2°).
Service Access	6.00	5.50	Maximum gradient of 8.5°
Remuck cubbies	6.00	5.50	15.0 m in length every 80 m
Cross-cut between Declines	6.00	5.50	13.25 m in length every 80 m
Mining Panel Access	6.00	5.50	Maximum gradient of 8.5°
Stockpile in Waste	6.00	5.50	15.0 m in length every 160 m
Stockpile in Mining Zones	6.00	5.50	15.0 m in length every 80 m
Ventilation	–	–	5.0 m in diameters depending on requirements
Ore Pass	–	–	3.0 m in diameters depending on requirements

The Kakula 2017 PEA development design is shown in Figure 24.31.

Figure 24.31 Kakula 2017 PEA Development

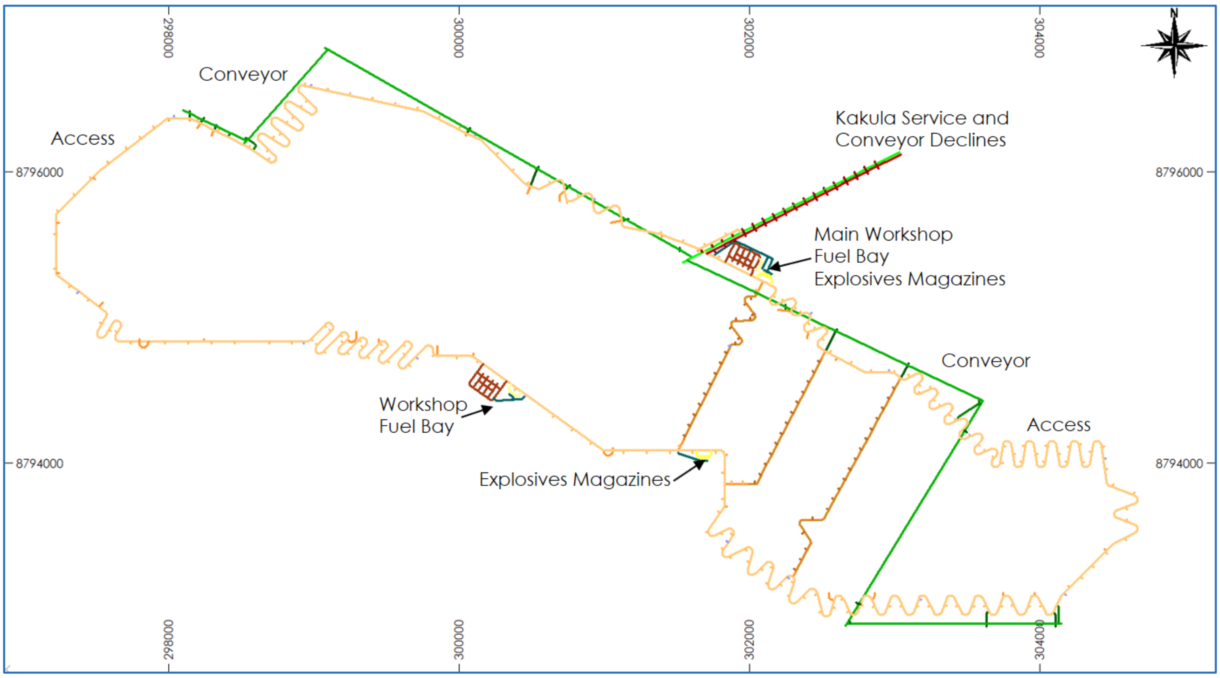

Figure by OreWin, 2017.

Selection of the mining method was dictated by mining height and dip. The controlled convergence room-and-pillar method was selected for heights greater than 3 m and less than 6 m and dip less than 25 degrees. The drift-and-fill with pastefill was selected for heights greater than 6 m. The drift-and-fill with pastefill method was also selected for heights greater than 3 m and less than 6 m and dip greater than 25 degrees.

Two drift-and-fill mining lifts were selected for the stope heights greater than 6.00 m. The maximum height of the drift-and-fill mining panel is 6.00 m and the minimum height of the drift-and-fill mining panel is 3.00 m.

Mine panels were also spilt by grade into primary and secondary stages for scheduling. Primary panels contain Cu>6.6% and secondary panels contain Cu<6.6% and Cu>3%. Figure 24.32 shows the location of the primary and secondary mining zones and the Kakula 2017 PEA 6 Mtpa development.

Figure 24.32 Kakula 2017 PEA Development and Mining Zones

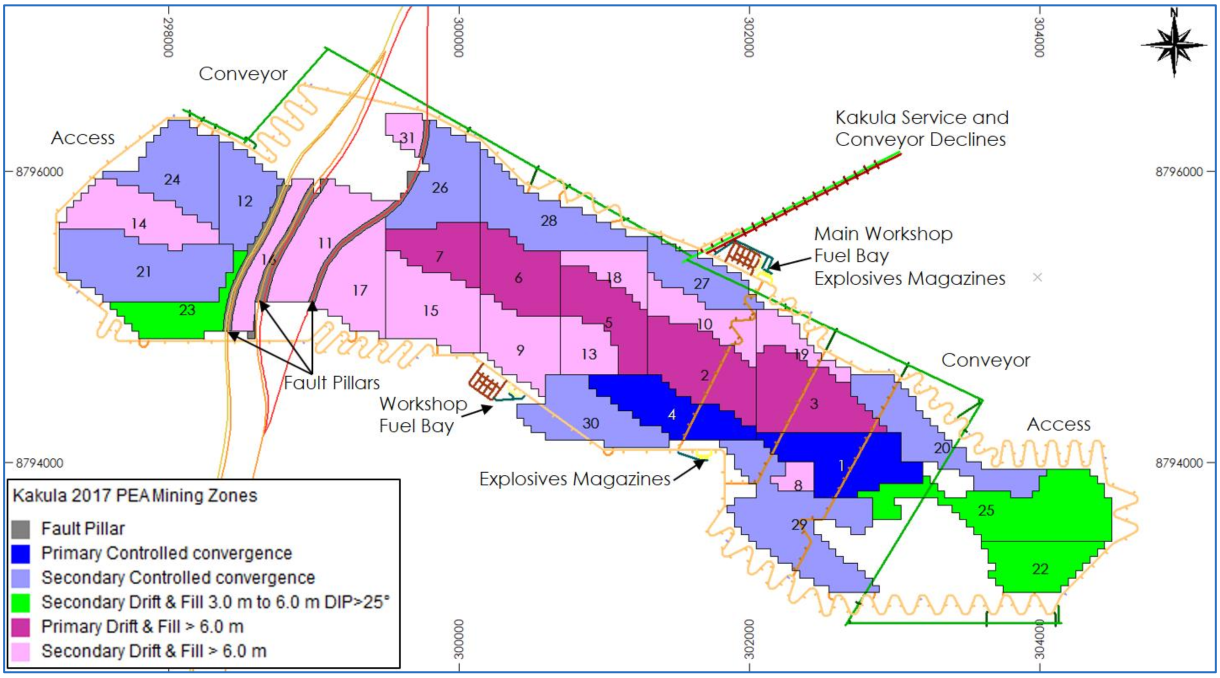

Figure by OreWin, 2017.

The following is a list of the key mining panels criteria and assumptions for the Kakula 2017 PEA design:

- Mining Zones 1 to 7 are the primary zones and zones 8 to 31 are secondary zones.
- Mining zones are defined by 20 m wide pillar boundaries. The pillar boundaries between two adjacent mining zones are 40 m in width.
- The primary access development within the mining zone consists of two 5.0 m wide parallel service drifts in 40 m wide pillar boundaries between the mining zones.
- The pillar between the mining stopes and the service drifts is 10 m. The pillar between two parallel service drifts is 10 m.
- The minimum panel size of the controlled convergence room-and-pillar mining method was assumed to be 125,000 square metres (500 m x 250 m).
- The minimum panel size of the drift-and-fill mining method was assumed to be 120,000 square metres (600 m x 200 m).
- The minimum and maximum panel height of the drift-and-fill mining method for each lift were assumed to be 3.00 m and 6.00 m respectively.
- The stope and production development of the drift-and-fill and controlled convergence room-and-pillar mining methods will be 10.0 m wide drifts.
- The drift-and-fill available heading for jumbo is 1 in 3 production heading.

24.5.4 Mining Costs

Mining costs were developed using the contractor mining costs from the current development at the Kansoko Mine, and factored fixed costs and unit rates from the Kamoa 2017 PFS.

24.6 Kakula 2017 PEA Processing

This section on recovery methods incorporates assumptions, analysis and findings of the Kakula 2017 PEA. It is proposed that the project proceeds with first completing the Kakula 6 Mtpa process plant, built in two stages of 2 x 3 Mtpa, followed by expansion for the Kamoa 6 Mtpa production.

24.6.1 Kakula Process Plant

The Kakula process plant consists of a 6 Mtpa Run-of-Mine (ROM) concentrator based on staged crushing, ball mill grinding and flotation. The plant design allowed the concentrator to be built into two phases in order to be aligned with the mine production schedule. Phase 1 will treat 3 Mtpa in line with the mine ramp up and the throughput will be doubled during Phase 2 to 6 Mtpa. The basis of design for the concentrator is outlined in Table 24.32. These availability figures are in line with industry norms for these types of operations after incorporating allowances for local issues such as power reliability.

Table 24.32 Kakula Process Plant Design Criteria

Production Information	Unit	Detail	Comment
Monthly Throughput	t/mo (d.b)	500,000	
Annual Throughput	t/a (d.b)	6,000,000	
Operating Days	days	365	
Overall Crusher Availability (design)	%	75	
Crusher Operating Time	h/a	6570	
Crushing Circuit Feed Rate	t/h	609	
Design Utilisation	%	87	
Run Hours	h/a	7598	
Plant Throughput	t/d	12 635	
Milling Trains installed	#	2	
Plant Throughput per Mill	t/h	263	
Maximum Feed Rate	t/h	290	+10%
Average Feed Grade	% Cu	5.76	Kamoa-Kakula 2017 PEA Mine Plan
Design Feed Grade	% Cu	8.07	Max Annual Average Grade (Years 1–10)
Concentrate Grade	% Cu	54.9	From testwork
Copper Recovery	%	87.9	From testwork
Design Mass Pull	Mass %	12.9	Based on Design Feed Grade
Collector Dosage	g/t milled	210	
Promoter Dosage	g/t milled	40	
Frother Dosage	g/t milled	91	
Flocculant Dosage	g/t milled	35	

24.6.1.1 Process Description

All underground plant feed sources will pass through a 250 mm square grizzly before being conveyed from the mine to surface stockpiles. A diverter is available at the surface to allow barren development rock to be stockpiled for removal and to allow stockpiling of material for later feeding to the ROM stockpile via an emergency bin as required. An over-belt magnet removes tramp steel from the material before it is sent to the ROM stockpile.

Variable speed apron feeders are available to recover material from the ROM stockpile at a desired tonnage and feed the crushing section.

The crushing plant utilises an open circuit primary crusher and secondary crushers, closed with screens, to generate a crusher product of minus 10 mm. All screens and crushers have dedicated feed bins with vibrating feeders controlling the feed rate into the crushers or screens.

The crushing plant has a primary double deck screen which classifies into oversize going to the primary crusher, middlings reporting to the secondary crusher and -10 mm final product which is conveyed to the mill feed stockpile.

Both primary and secondary crusher product feed a common conveyor which feeds double deck sizing screens (top deck relieving). Sizing screen oversize recycles back to the secondary crusher, whilst screen undersize reports to final crusher product. The combined primary screen and sizing screen undersize is sent to the mill feed stockpile.

The mill feed stockpile has two tunnels with apron feeders. Each tunnel supplies a dedicated feed to the two primary ball mills. Each mill feed conveyor has a grinding ball feeding system with automatic addition control.

The primary ball mill is designed to produce a coarse product of P_{80} 150 to180 µm. The primary ball mills receive dry new feed together with primary cyclone underflow. Mill discharge is pumped to a classification cyclone with undersize being recycled to the primary mill and cyclone overflow reporting to the secondary mill discharge sump.

The secondary mill section has a ball mill, identical to the primary mill. Mill discharge is cycloned and cyclone underflow reports back to the mill feed while cyclone overflow (grind of P_{80} 53 µm) reports to the trash screening section before feeding the rougher flotation section.

The rougher flotation section has a conditioning tank for reagent addition (frother and collector) before slurry is pumped to rougher-scavenger flotation banks (one bank per grinding line). Rougher concentrate from the first cells in each train are pumped to a common rougher cleaning bank. Rougher tails pass directly to the scavenger float cells.

Scavenger concentrate from both banks forms the majority of the regrind mill feed while the scavenger tails streams report to the tailings thickener for TSF disposal.

Rougher cleaner concentrate is sent to rougher recleaner flotation and a coarse final concentrate is produced. The rougher recleaner concentrate is pumped to the concentrate thickener. Tails from both the rougher cleaner and rougher recleaners are sent to the regrind milling section.

The three regrind mill feed streams, scavenger concentrates, rougher cleaner tails and rougher recleaner tails, are pumped to the regrind feed tank. Regrind circuit feed is pumped to the regrind densifying cyclones, with cyclone underflow reporting to the regrind mill. Cyclone overflow (P_{80} 10 µm or finer) reports directly to the regrind circuit product.

Reground slurry joins the desifying cyclone overflow and is pumped to the scavenger cleaners. Scavenger cleaner concentrate reports to the scavenger recleaners. Both scavenger cleaner tailings and recleaner tailings join with scavenger tailings and report to final tailings thickening.

Scavenger recleaner concentrate is combined with rougher recleaner concentrate into the final concentrate thickener.

Thickened concentrate is fed to a final concentrate pressure filter system, which subsequently feeds a bagging plant. The concentrate is sampled before bagging and despatch to customers.

Final tailings thickener underflow is pumped via a tailings tank to the TSF.

The block flow diagram for the crushing and milling circuit is shown in Figure 24.33. The block flow diagram for flotation, concentrate handling and tailings is shown in Figure 24.34.

Figure 24.33 Kakula Crushing and Milling

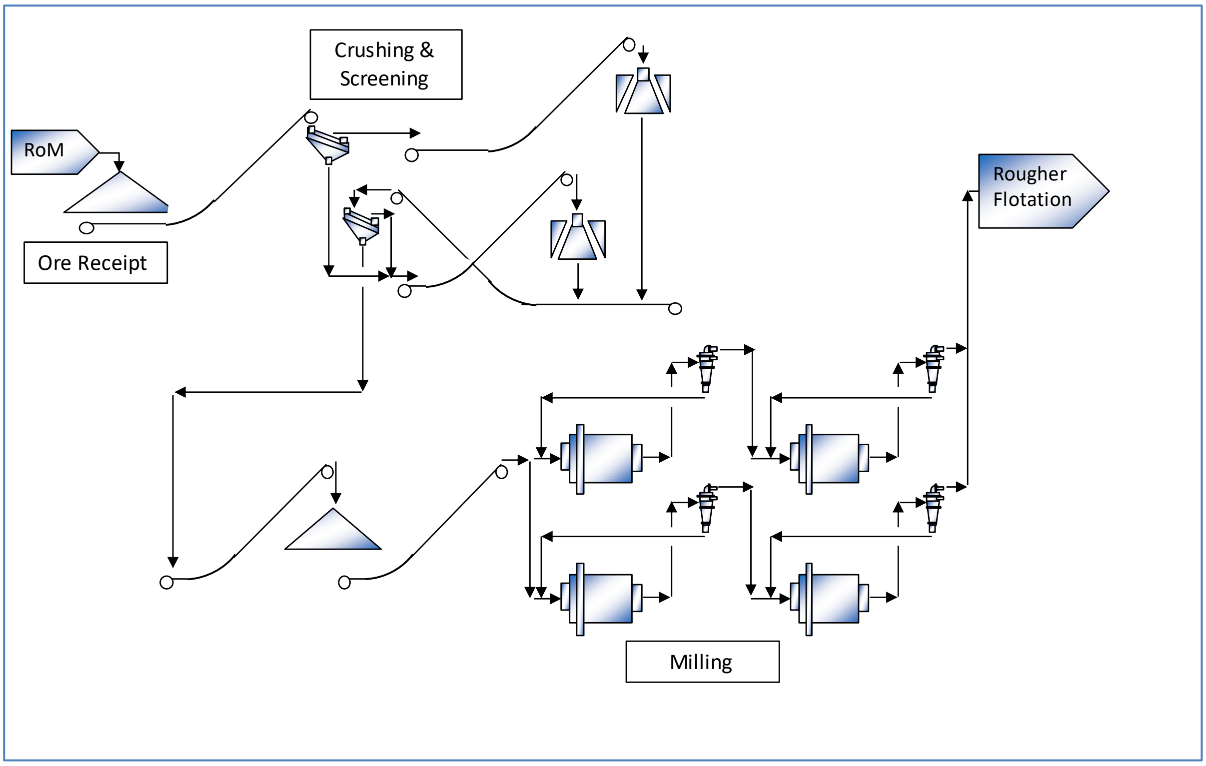

Figure courtesy MDM, 2017.

Figure 24.34 Kakula Flotation Circuit

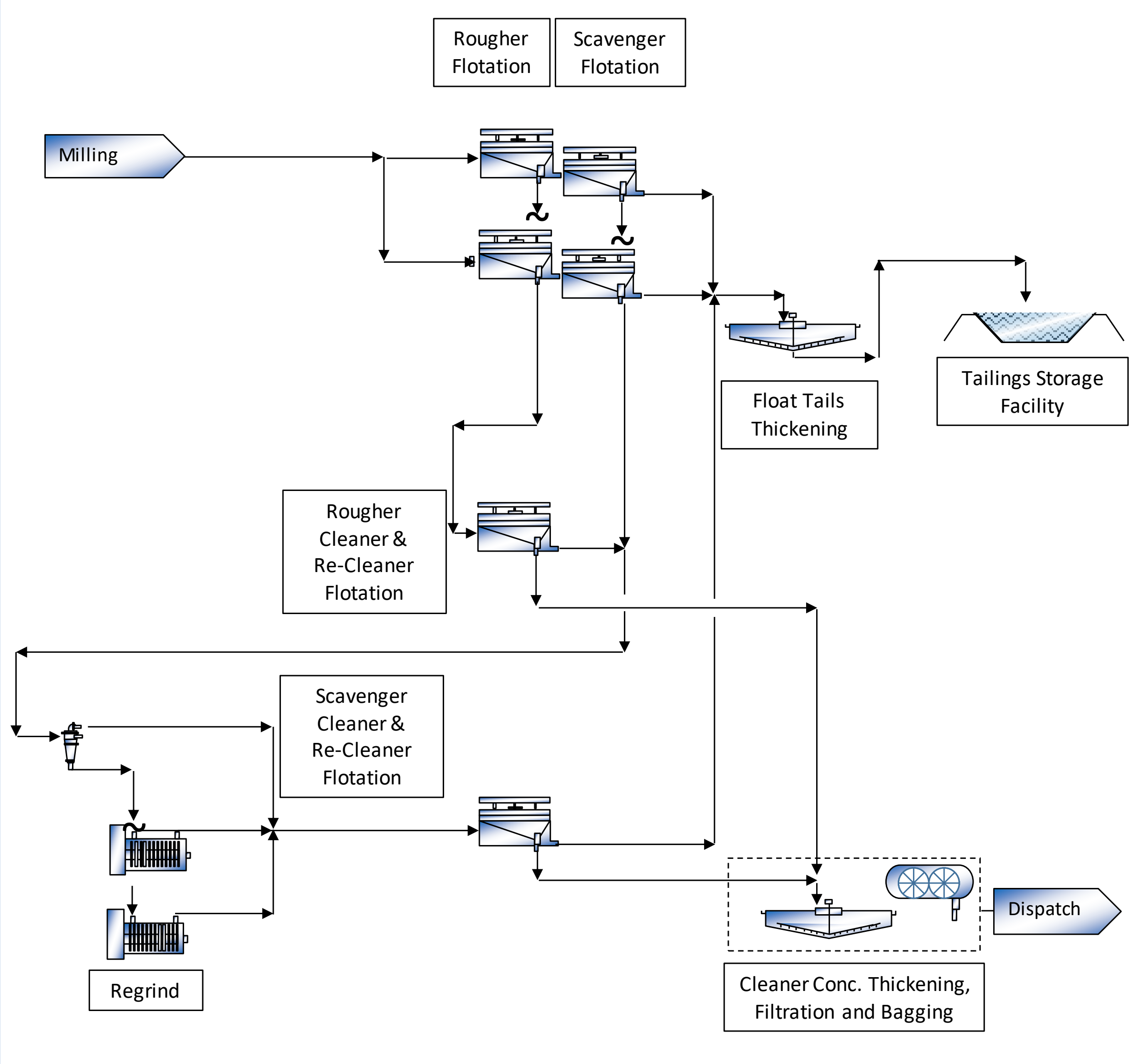

Figure courtesy MDM, 2016.

24.6.1.2 Reagents, Services and Utilities

Reagent plants, located close to the flotation circuit, provide for the mixing and supply of the necessary reagents for flotation and flocculants for thickening.

All flotation cells are forced air and dedicated blowers supply manifold air for the flotation cells.

Raw water from a wellfield is pumped to a raw water pond from which water is transferred to the plant. Filtration and treatment plants use the raw water to produce a range of water qualities as required for potable, gland seal, cooling, fire and process water usage. Distribution systems for each water type are included, ensuring delivery of sufficient quantity at the required pressure in each instance.

Compressed air is supplied and distributed for general plant requirements and filter presses. A dried air supply is available for air actuated instruments and valves.

24.6.1.3 Concentrator Equipment

Table 24.33 provides a summary of the major mechanical equipment for the proposed concentrator. This list forms the basis of a much more detailed concentrator capital cost estimate.

Table 24.33 Kakula Concentrator Equipment Requirements Summary

Item	Description	Size/Capacity	No. Required + standby	Power Installed kW per unit
Crushers	Primary cone	CS660	2	315
	Secondary cone	CH865	3	500
Screens	Primary	2.4 m x 4.27 m	2	45
	Secondary	3.1 m x 6.1 m	4	55
Mills	Primary Ball Mill	22 ft x 36 ft	2	7,000
	Secondary Ball Mill	22 ft x 36 ft	2	7,000
	Concentrate regrind	IsaMill M10000	2	3,000
Cyclones	Primary cluster	750 mm Diameter	4 + 1	500 (1+1 feed pump)
	Secondary cluster	420 mm Diameter	9 + 1	355 (1+1 feed pump)
	Concentrate regrind cluster	165 mm Diameter	15 + 1	75 (1+1 feed pump)
Blowers	Flotation air	65 700 Nm^3/h @ 150 kPa	4 + 1	200
Flotation cells (includes agitators)	Rougher	320 m^3	4	280
	Scavenger	320 m^3	10	280
	Rougher cleaner	50 m^3	5	75
	Rougher recleaner	30 m^3	6	45
	Scavenger cleaner	160 m^3	6	160
	Scavenger recleaner	30 m^3	6	45
Thickeners	Concentrate	20 m Diameter	1	11
	Tailings	50 m Diameter	1	18
Filters	Concentrate	Hyperbaric Disc 57216-2L	2	55
Tailings Pumps	Centrifugal	840 m^3/h	4 + 4	185

Table 24.34 lists the estimated projected water, consumables, and power requirements for the concentrator.

Table 24.34 Kakula Projected Concentrator Water, Power, and Consumables

Item	Description	Units	Consumption per tonne of Plant Feed	Annual Requirement
Power	Electric	kWh	58.8 kWh/t	353 GWh
Water	Raw make-up	m^3	0.5 m^3/t	3,025 ML
Reagents	Frother	g/t	95 g/t	570 t
	Collector	g/t	156 g/t	936 t
	Promoter	g/t	28 g/t	168 t
	Flocculant (Tailings and Concentrate)	g/t	35 g/t	210 t
Consumables	Grinding media (75 mm steel balls)	kg/t	0.82 kg/t	4,920 t
	Grinding media (35 mm steel balls)	kg/t	0.82 kg/t	4,920 t
	Grinding media (2 mm Ceramic)	g/t	68 g/t	408 t

Most consumables are supplied in bulk bags or containers. A kibble is used to load grinding media into the ball mills. The low abrasion index (Ai = 0.08) ensures that ball consumption will be relatively low compared to most similar projects.

24.6.1.4 Capital and Operating Costs

Capital costs are defined as the expenditure required during the design, construction and commissioning phases of the project. This includes all costs associated with labour, construction, plant and equipment, bulk materials, other materials, permanent equipment, sub-contracts, packaging, transportation, loading, off-loading, strategic spares and capital indirect costs which contribute to the physical construction of the project.

The Kamoa-Kakula 2017 PEA analysed a two-phase sequential expansion of production to 12 Mtpa from the proposed Kakula 6 Mtpa PEA at the Kakula Deposit and also the Kansoko Mine at the adjacent Kamoa deposit. The mines continue to ramp up to 12 Mtpa combined by Year Nine. Once the Kansoko and Kakula Mines near the end of their life, Kamoa North comes on line to maintain the overall production at 12 Mtpa.

The costing for the concentrator allows for the 3 Mtpa portion of the Kamoa-Kakula Phase 1 and a 3 Mtpa Phase 2. The phased approach allows for a two-year delay prior to the commencement of Phase 2, ensuring that the mining plan vs. capital expenditure is optimised with regards to cash flow.

The following inputs and documents were identified and used in compiling the capital cost estimate:

- Process design basis.
- Site plot plans.
- Block flow diagrams.
- Process flow diagrams.
- Mechanical equipment list.
- Battery limits as described in the study documentation.

Costs have been estimated for the following disciplines:

- Earthworks.
- Civil works.
- Structural steel fabrication, supply and erection.
- Platework fabrication, supply and erection.
- Mechanical equipment supply.
- Mechanical equipment installation.
- Pipework fabrication, supply and erection.
- Electrical and C&I supply and erection.
- Transportation to site.
- EPCM services.
- First fills and spares.
- Infrastructure.

The operating cost estimate includes the fixed (labour and maintenance) costs and variable costs components (reagents, grinding media and power costs). The operating cost figure excludes rehabilitation, mining, insurance costs, import duties and all other taxes.

The sources of information and assumptions are as follows:

- Vendor information and quotations based on the Kamoa 2016 PFS study, factored.
- Plant labour rates and staffing levels as supplied by Kamoa Copper SA.
- Power cost supplied by Kamoa Copper SA.
- MDM Technical Africa (Pty) Ltd (MDM) knowledge and experience.

24.6.2 Kakula 2017 PEA 6 Mtpa Processing

Run-of-mine material will be fed to an onsite concentrator via conveyor to produce a saleable copper concentrate. The concentrator will comprise two parallel 3 Mtpa plants to process 6 Mtpa of plant feed at full production.

Each 3 Mtpa plant in its first year of operation ramps up to full capacity after a period shown in Table 24.35. Plant 1 will begin processing material at the beginning of July Year 1 and run standalone until the June Year 3. At the beginning of July Year 3 Plant 2 will begin processing material and from Year 4 onwards both plants will be processing material at a total combined capacity of 6 Mtpa.

Table 24.35 Kakula Process Plant Ramp-up Schedule

Month no.	Plant 1 - 3 Mtpa (kt)	Plant 2 - 3 Mtpa (kt)
1	101	225
2	162	251
3	200	251
4	225	251
5	237	251
6	245	251
7	251	251
8	251	251
9	251	251
10	251	251
11	251	251
12	251	251
Total	**2,672**	**2,981**
13+	251	251

24.6.3 Kamoa 2017 PFS Process Plant

The Kamoa 2017 PFS process plant consists of a 6 Mtpa Run-of-Mine (ROM) concentrator based on staged crushing, ball mill grinding and flotation. The plant design allowed the concentrator to be built into two phases in order to be aligned with the mine production schedule. Phase 1 will treat 3 Mtpa in line with the mine ramp up and the throughput will be doubled during Phase 2 to 6 Mtpa. The basis of design for the concentrator is outlined in Table 24.36.

Table 24.36 Kansoko Process Plant Design Criteria

Option	Units	Value	Comment
Flotation Feed	Mtpa	6	–
Average Feed Rate	t/h	790	–
Maximum Feed Rate	t/h	869	+10%
Average Feed Grade	% Cu	3.81	Kamoa PFS Mine Plan
Early Ore Grade	% Cu	4.26	Average (Years 1–10)
Design Feed Grade	% Cu	5.14	Max Annual Average Grade Years 1–10 plus 13%
Relative Abundance - Hypogene (%)	Mass %	89	Kamoa PFS Mine Plan
Relative Abundance - Supergene (%)	Mass %	11	Kamoa PFS Mine Plan
Concentrate Grade	% Cu	37	Lower than mine production assumption, see text
Copper Recovery	%	86	From testwork
Design Mass Pull	Mass %	11.9	Based on Design Feed Grade

These availability figures are in line with industry norms for these types of operations after incorporating allowances for local issues such as power reliability.

The ability to blend feed from multiple sources underground should provide the project with a high degree of control over plant feed grade and as such the maximum head grade expected has been chosen to be only marginally higher than the highest annual average grade.

The concentrate grade of 37% Cu was chosen based on a high chalcopyrite feedstock and represents the most conservative case in terms of tonnes to be thickened, filtered and bagged. A grade of 39% Cu has been used in mine planning and this is a legitimate average grade based on testwork results.

The process plant design fundamentals for Kansoko are identical to the Kakula design relative to process flow routes. The only deviations relate to differences in mass balances and associated equipment sizes as a result of changes in copper head grade and associated recoveries.

The Kansoko block flow diagram is similar to the Kakula block flow diagram.

The Kansoko reagents, services and utilities plants are identical to the Kakula plant.

Table 24.37 provides a summary of the major mechanical equipment for the proposed concentrator. This list forms the basis of a more detailed concentrator capital cost estimate.

Table 24.37 Kansoko Concentrator Equipment Requirements Summary

Item	Description	Size/Capacity	No. Required + standby	Power Installed kW per unit
Crushers	Primary cone	CS660	2	315
	Secondary cone	CH865	3	500
Screens	Primary	2.4 m x 4.27 m	2	45
	Secondary	3.1 m x 6.1 m	4	55
Mills	Primary Ball Mill	22 ft x 36 ft	2	7,000
	Secondary Ball Mill	22 ft x 36 ft	2	7,000
	Concentrate regrind	IsaMill M10000	2	3,000
Cyclones	Primary cluster	750 mm Diameter	4 + 1	500 (1+1 feed pump)
	Secondary cluster	420 mm Diameter	9 + 1	355 (1+1 feed pump)
	Concentrate regrind cluster	165 mm Diameter	15 + 1	75 (1+1 feed pump)
Blowers	Flotation air	65 700 Nm^3/h @ 150 kPa	4 + 1	200
Flotation cells (includes agitators)	Rougher	320 m^3	4	280
	Scavenger	320 m^3	10	280
	Rougher cleaner	50 m^3	5	75
	Rougher recleaner	30 m^3	6	45
	Scavenger cleaner	160 m^3	6	160
	Scavenger recleaner	30 m^3	6	45
Thickeners	Concentrate	20 m Diameter	1	11
	Tailings	50 m Diameter	1	18
Filters	Concentrate	Hyperbaric Disc 57216-2L	2	55
Tailings Pumps	Centrifugal	840 m^3/h	4 + 4	185

Table 24.38 lists the estimated projected water, consumables, and power requirements for the concentrator.

Table 24.38 Kansoko Projected Concentrator Water, Consumables, and Power

Item	Description	Units	Consumption per tonne Plant Feed	Annual Requirement
Power	Electric	kWh	58.8 kWh/t	353 GWh
Water	Raw make-up	m^3	0.5 m^3/t	3,025 ML
Reagents	Frother	g/t	95 g/t	570 t
	Collector	g/t	156 g/t	936 t
	Promoter	g/t	28 g/t	168 t
	Flocculant (Tailings and Concentrate)	g/t	35 g/t	210 t
Consumables	Grinding media (75 mm steel balls)	kg/t	0.82 kg/t	4,920 t
	Grinding media (35 mm steel balls)	kg/t	0.82 kg/t	4,920 t
	Grinding media (2 mm Ceramic)	g/t	68 g/t	408 t

Most consumables are supplied in bulk bags or containers. A kibble is used to load grinding media into the ball mills. The low abrasion index of (Ai = 0.08) ensures that ball consumption will be relatively low compared to most similar projects.

24.6.3.1 Concentrator Capital and Operating Costs

Capital costs are defined as the expenditure required during the design, construction and commissioning phases of the project. This includes all costs associated with labour, construction, plant and equipment, bulk materials, other materials, permanent equipment, sub-contracts, packaging, transportation, loading, off-loading, strategic spares and capital indirect costs which contribute to the physical construction of the project.

The operating cost estimate includes the fixed (labour and maintenance) costs and variable costs components (reagents, grinding media and power costs). The operating costs are expressed in United States Dollar (USD) per tonne milled. Overall operating cost for 4 Mtpa Kansoko process plant is detailed in Section 24.3. The operating cost figure excludes rehabilitation, mining, insurance costs, import duties, and all other taxes. Capital and operating costs are summarised earlier in this section.

The sources of information and assumptions for cost calculations are as follows:

- Vendor information and quotations based on the PFS study, factored.
- Plant labour rates and staffing levels as supplied by Kamoa Copper SA.
- Power cost supplied by Kamoa Copper SA.
- MDM Technical Africa (Pty) Ltd (MDM) knowledge and experience.

24.6.4 Comments on Section 24.6

ROM from both Kakula and Kansoko mines is assumed to have a topsize of 300 mm controlled by intensive blasting and 250 mm square grizzly installations at each dump point underground. If this top size control is found to be unmanageable, then additional underground crushing may be required. Note that underground grizzly sizes can only be relaxed with caution as particles larger than 300 mm are likely to cause problems for the conveying system that brings the material to the surface from underground.

The flotation circuit configuration recommends not using recycle streams in accordance with the XPS testing philosophy. Flowsheet provisions for possible recycles should be considered in the next project phase.

The copper mineralisation determines how much copper is recoverable by flotation and the grade of concentrate that can be generated. Kamoa mineralisation is variable and further work is needed to better define mineralogy in the various parts of the deposit.

24.7 Kakula 2017 PEA Infrastructure

The infrastructure for the Kakula 2017 PEA must support two separate processing plants, one at Kakula and one at Kansoko. The project infrastructure includes power supply, tailings dams, communications, logistics, transport options, materials handling, water and waste water, buildings, accommodations, security, and medical services.

The overall site plan is shown in Figure 24.35. The Kansoko plant area is shown in Figure 24.37 and the Kakula plant area is shown in Figure 24.36. All associated infrastructure to operate the concentrator and mine at Kakula and Kansoko have been allowed for.

Figure 24.35 Kansoko and Kakula Combined Site Layout

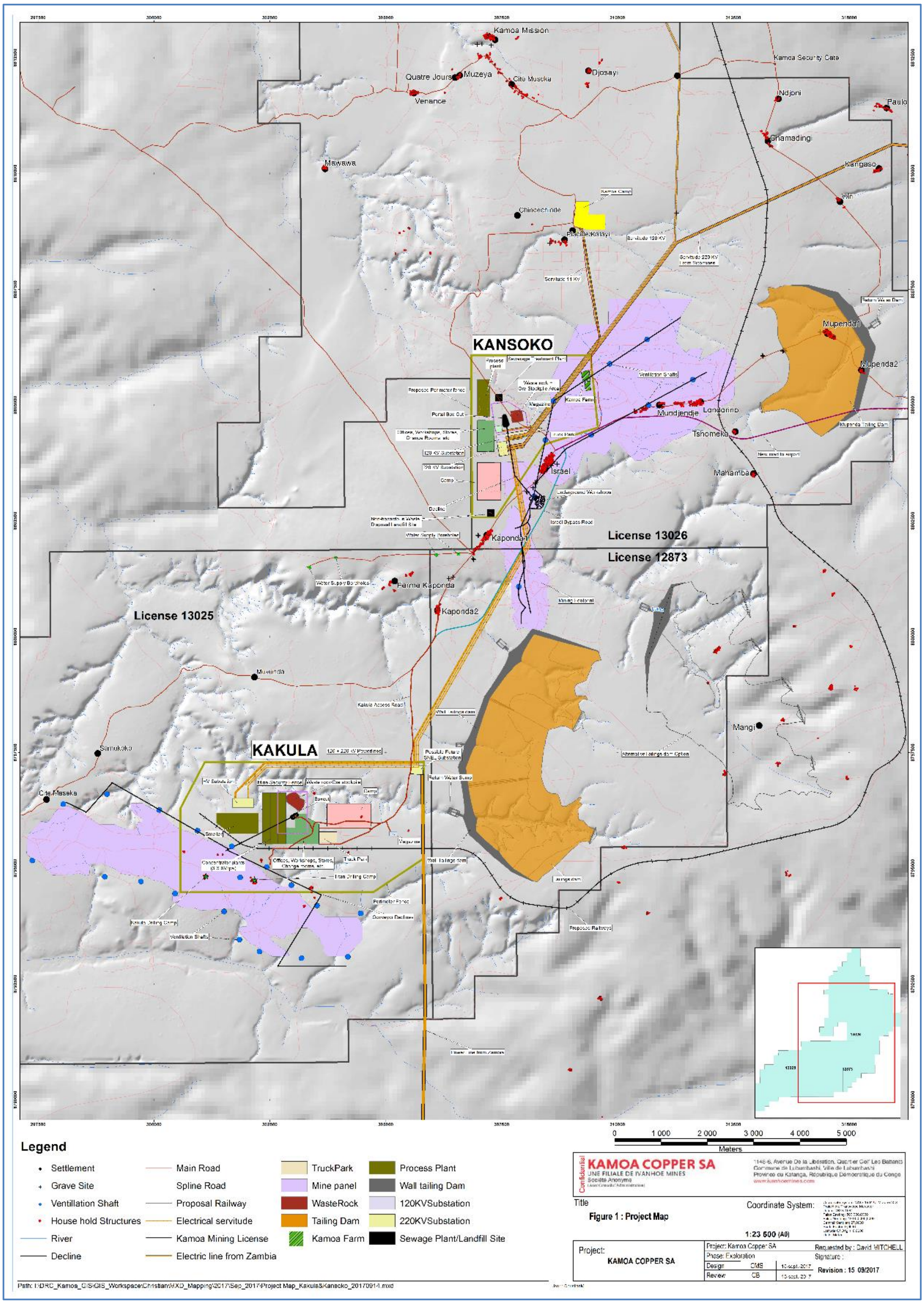

Figure by Kamoa Copper SA, 2017.

Figure 24.36 Kakula Site Conceptual Concentrator

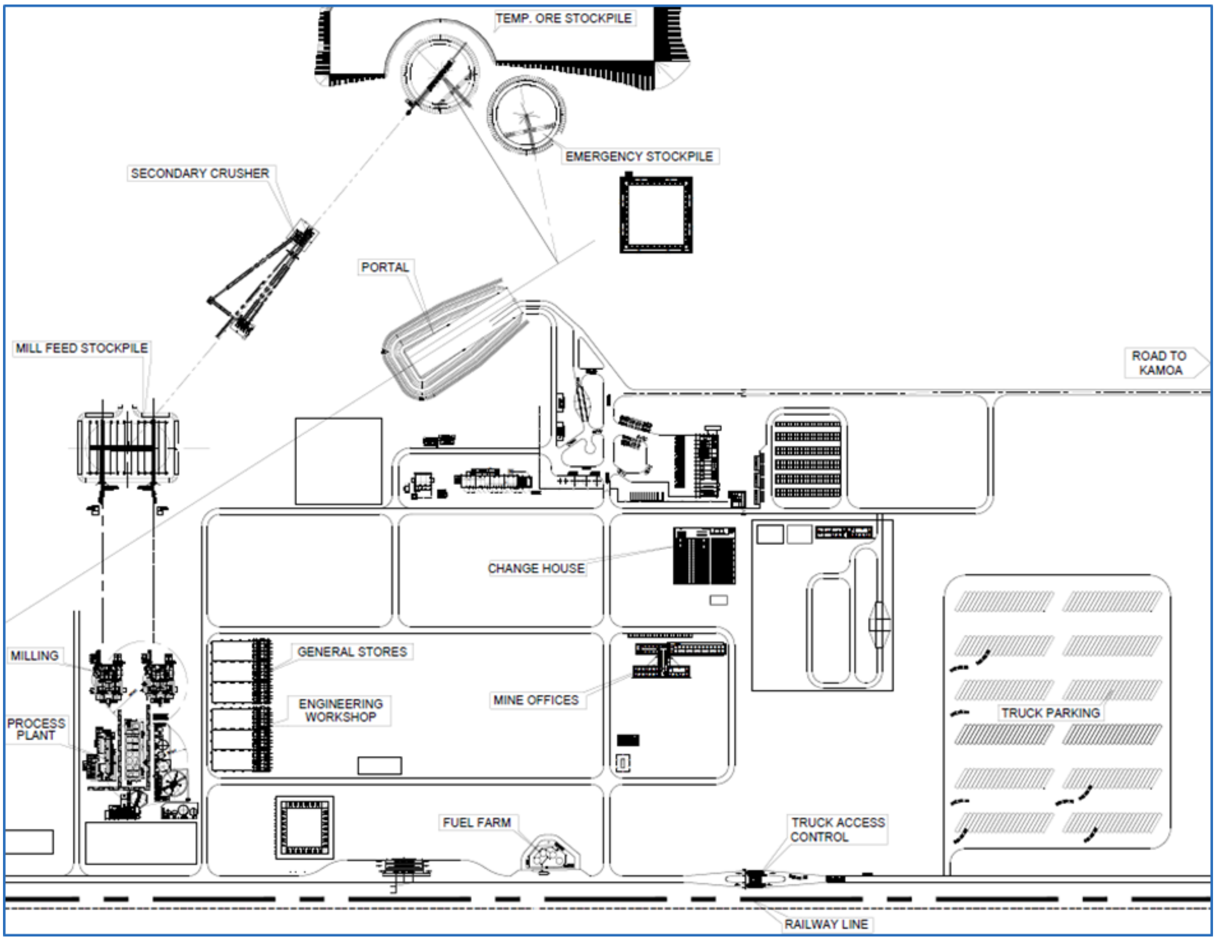

Figure By MDM, 2017.

Figure 24.37 Kansoko Site Conceptual Concentrator

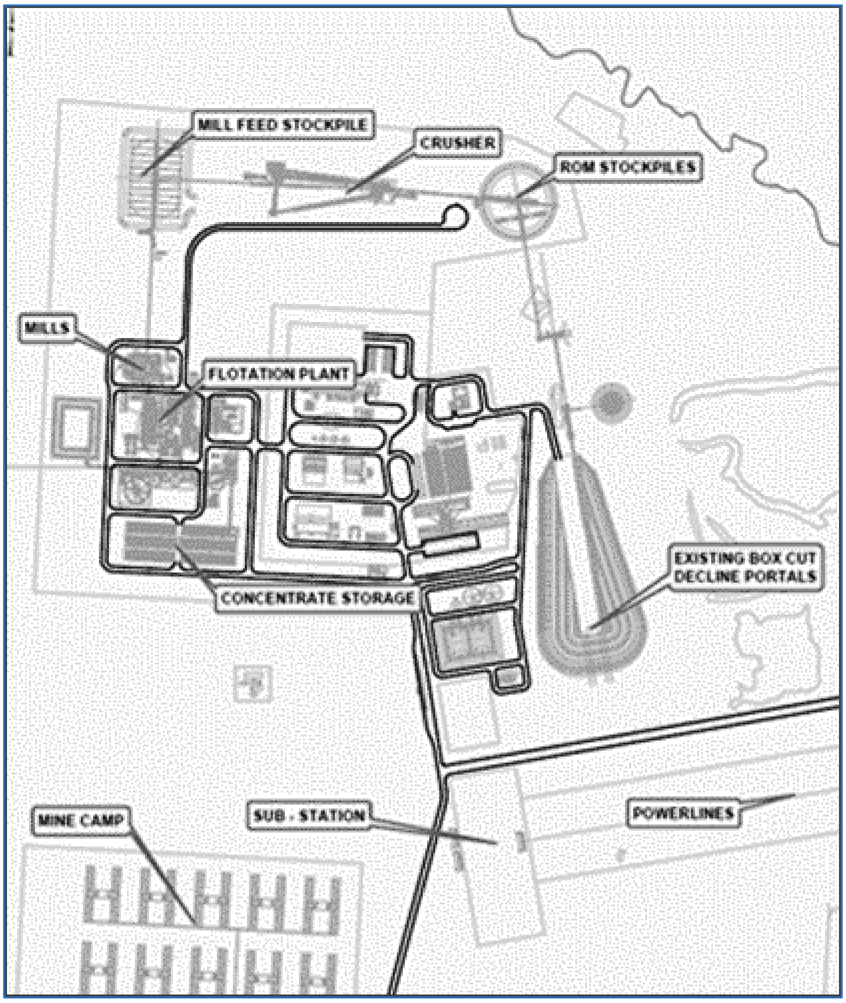

Figure by MDM, 2017.

24.7.1 Power

Power for the Kamoa-Kakula Project is planned to be sourced from the DRC's state-owned power company (SNEL, Société Nationale d'Electricité) electrical interconnected grid.

The supply will feed into the new 220/11 kV substations at each mine from where the process plants and mines will receive power. Power reticulation to vent shafts, TSF, camps etc. would be via 11 kV overhead lines. For construction power (10 MW), a 120 kV high-voltage line (20 km) has been built from a point from the RO-Kisenge line to the Kansoko mine. The line would be extended to the Kakula Mine. Diesel generators would be installed to provide standby power during construction and operation.

There are currently High Voltage (HV) power lines installed to the Kansoko site and will be upgraded according to the project requirements. Power lines crossing roads are evident and will be inspected as per the future required abnormal load delivery requirements.

24.7.2 Kakula 2017 PEA TSF

Epoch Resources (Pty) Ltd (Epoch) prepared a study of potential the tailings storage facilities (TSF) at Kakula and Kansoko. The Kansoko plant will utilise the same TSF as per the Kamoa 2017 PFS (Mupenda), and the Kakula plant will use its own dedicated TSF adjacent to the plant site. The following items were considered:

- A high-level site selection study to identify possible sites for the TSF.
- A TSF that accommodates the required tonnes of tailings for the LOM. The Kansoko Mupenda TSF is based on the Kamoa 2017 PFS, and has not been updated to accommodate the proposed 6 Mtpa plant throughput. This will be done during the next stage of the study.
- A Return Water Sump (RWS) associated with the TSF.
- The associated infrastructure for the TSF (i.e. perimeter slurry deposition pipeline, stormwater diversion trenches, perimeter access road etc.).

A site selection study was undertaken to identify a preferred site for the Kakula TSF. A modelling exercise was undertaken, whereby sites were identified based on the topography of the site and their ability to contain the required tailings. Six new sites were identified (A, B, C, D, E, and F) and TSF 8 in Figure 24.37, which was identified during the Mupenda PFS site selection study, was also included as a potential site for Kakula. The proposed TSF positions were reviewed, and option D was selected due to it having the lowest capital cost, least environmental impact and least amount of lining required. The staged development of the TSF after Year 1 and after Year 26 of operation (end of Kakula project life) is shown in Figure 24.38.

Figure 24.38 General Topography of the Kakula TSF Area

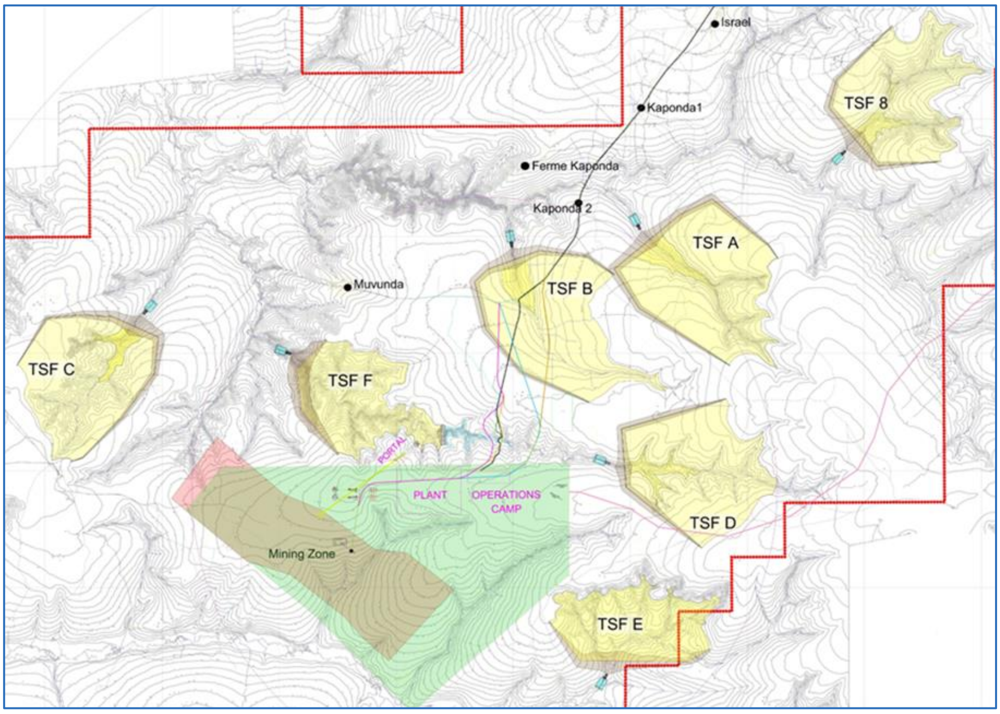

Figure by Kamoa Copper SA, 2016.

Figure 24.39 Kakula TSF Layout

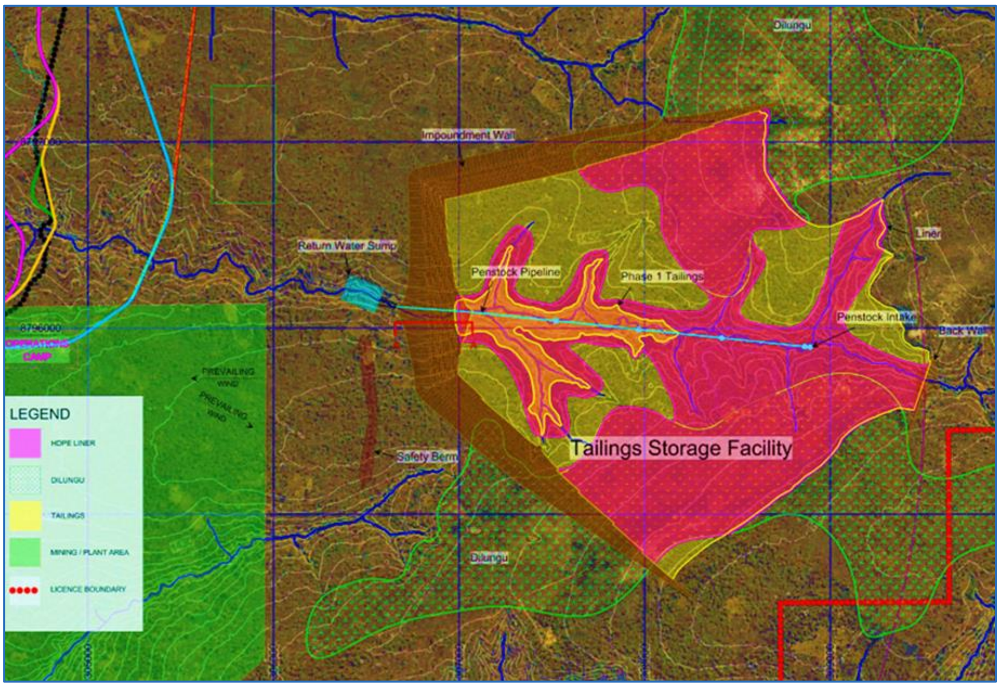

Figure by MDM, 2016.

24.7.3 Site Access and Transport

A reliable and safe 42 km main access road from Kolwezi airport to the Kansoko and Kakula project have been allowed for. The proposed new road sections are shown in Figure 24.40.

Figure 24.40 Proposed Access

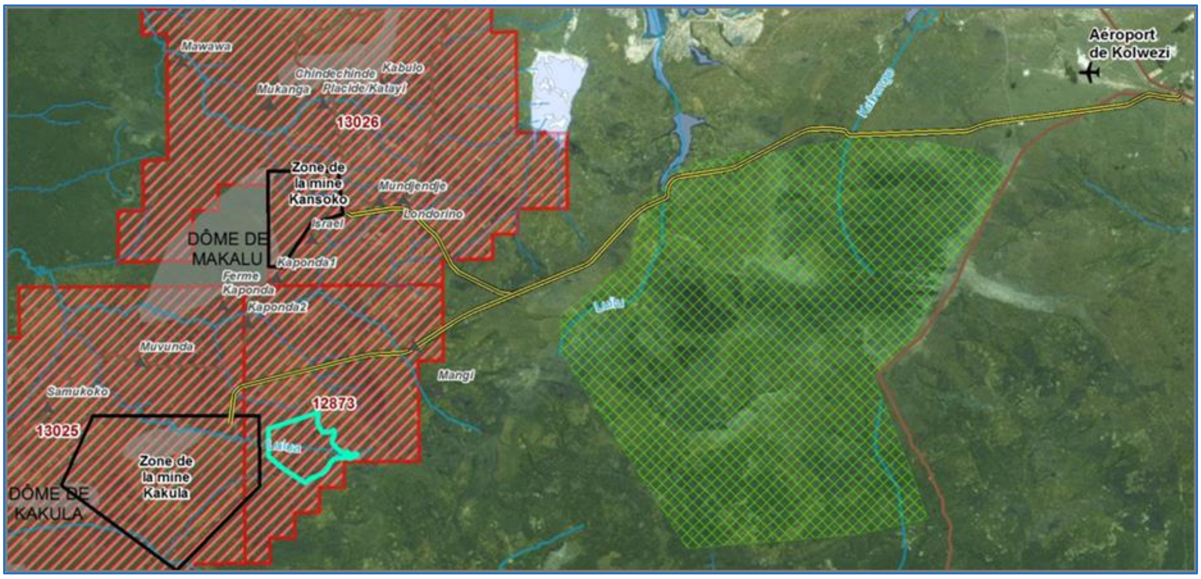

Figure by MDM, 2016.

The new road is to be gravel and will be built up substantially to achieve the necessary drainage. The road will be 9 m wide with a safe driving speed of 80 km/h. There is upgrade potential to use this base and black top surface the road in future.

The following facilities have been allowed for inside the plant and mine areas:

- Plant roads.
- Plant to portals roads.
- Plant to tailings storage facilities.
- Service roads (conveyor, ventilation fans, slurry pipelines).
- Camp access and internal roads.

A preliminary bridge assessment has been undertaken. No constraints have been identified to date. Costing has been allowed for a new access road from Kolwezi airport to Kansoko and Kakula plants. This road will be able to service the construction and 6 Mtpa plant requirements. With the development full 12 Mtpa capacity, the rail link will support the increase in logistics requirements.

A phased logistics solution is proposed for the Kakula 2017 PEA as per the Kamoa 2017 PFS. Initially the corridor between southern DRC and Durban in South Africa is viewed as the most attractive and reliable export route using trucks to Ndola in Zambia and rail from there to the port of Durban. As soon as the railroad between Kolwezi and Dilolo, a town near the DRC-Angolan border, is rehabilitated, production is expected to be transported by rail to the port of Lobito in Angola.

Costs have been allowed for a 40 km rail link from the existing Lobito rail infrastructure to Kansoko and Kakula. It is currently planned to receive and deliver goods by truck and thereafter utilise the rail link which will substantially reduce overall transport cost. See Figure 24.41

Figure 24.41 Rail from Kakula and Kansoko to Existing Lobito Infrastructure

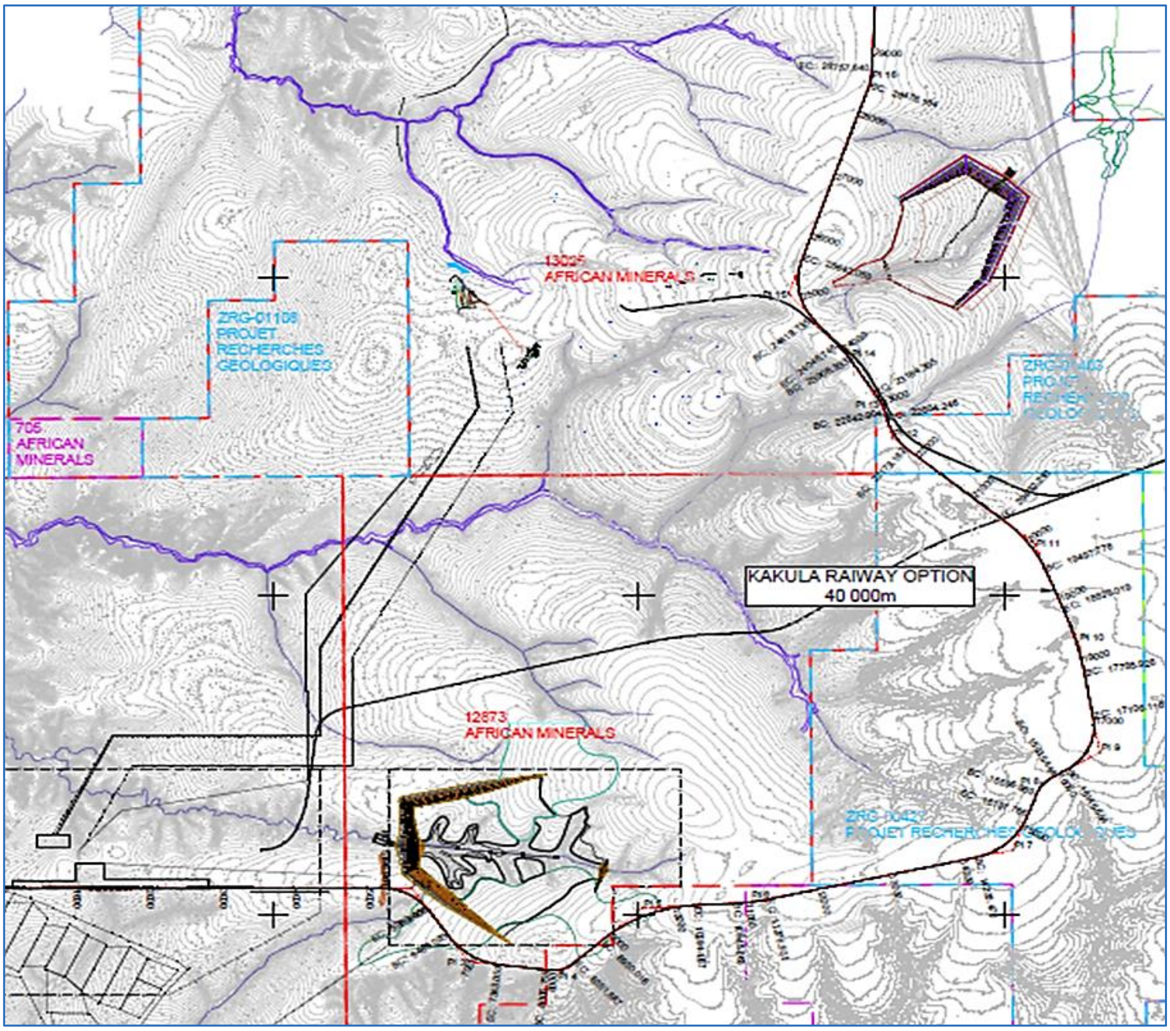

Figure by MDM, 2016.

Freight forwarding contractors will be appointed for the international component of the logistics requirements. A local DRC customs clearing/broker partnership will be established. Applicable duty-free goods will be transported to bonded laydown yards as required. Central warehousing facilities will be set up, to consolidate transport loads and to ensure that bonds are not retained on shipping containers.

During the operational phase, reagents and consumables will be sourced and transported from South Africa, DRC and/or neighbouring countries. The routing of reagents and consumables to Kamoa-Kakula will be the subject of a future, separate transport study.

Lubumbashi international airport and the domestic Kolwezi airport will be utilised as specified in the Kamoa 2017 PFS. Currently all commercial flights land in Lubumbashi with a connecting flight to Kolwezi.

24.7.4 Water Supply

Raw water will be provided to the site via production boreholes, mine dewatering boreholes and mine decline dewatering. This will provide all necessary raw water which will then be used to provide the required process water makeup, gland water, fire and reagent make-up water. A return water pipeline will bring water from each TSF to the associated process water tank for re-use. Due to the high annual rainfall, local dams and rivers and mine dewatering, ample water is available to satisfy the required water demand for both plants.

Two potential sources have been identified for back-up bulk water supply:

- The aquifer within the sandstone forming the Kamoa and Makalu Domes which are the footwall to the mining operations.
- The Haute Luilu Dam approximately 13 km to the east of the plant area. This is an existing dam constructed in 1978 as a clean water diversion dam to prevent water in this tributary of the Luilu River flowing into the mining areas of Kolwezi. This dam is owned by Gecamines. The water is not used presently, and Gecamines has provided Kamoa Copper with written permission to use the water.

It is envisaged that all raw water can be supplied from the available ground water sources.

Potable water for local villages is currently obtained from local rivers and streams. Potable water for any future mining operation will be sourced from bore holes. Potable water for ablution facilities, kitchens and emergency stations (eyewash and showers) will be obtained from the bulk water system and treated by means of disinfection only (chlorination). An appropriate drinking water standard will be applied, referencing indicators such as bacterial content, residual chlorine, turbidity, and dissolved solids.

24.7.4.1 Stormwater and Wastewater

A stormwater management plan will be developed including pollution control dams and the pipelines with their required pumps are all based on the required DRC regulations.

Sewage from kitchens and ablutions will drain via underground sewers to a sewage treatment plant and treated to produce an effluent of a suitably safe standard for process use. Floor washings that contain organic contaminants, from kitchens and ablution blocks, will also drain via the sewers to the treatment plant. Oil traps within the plant workshops will be installed and the oil contained and recycled. Other wastewater streams and by products are recovered under the plants process design.

24.7.5 General Infrastructure

Fuelling infrastructure has allowed for separate fuelling areas for the concentrator and mining fleet at the Kakula and Kansoko sites. Fuel is currently being delivered to the Kansoko mine, and it is a reliable supply.

On site workshops have been allowed for to facilitate repair the mobile machinery on site. If vehicles break down on route to site, commercially-owned breakdown rigs with a towing capacity of up to 30 t are available.

Within the infrastructure costing, allowance has been made for camp, together with plant and perimeter security fencing. The fence follows a maintenance laterite access road providing patrol and fence maintenance access. Security control buildings at major access control points have been allowed for, including ablution facilities.

The fire protection and detection systems for the surface plant and infrastructure (excluding all underground mining which is covered separately) will be developed in consultation with, and subject to final approval from, the Owner’s risk assessors. The system will be designed to comply with DRC legislation, the project Health and Safety standard/s, project specifications and fire protection standards as adopted by the Project.

The clinic and first-aid facility will be housed together at a suitable position near the main gate. Medical equipment, including an ambulance, will be provided. Medical evacuation for ex-patriate employees will be provided by an outside contracting service.

Permanent villages called the Owners Camps, capable of accommodating 1500 persons each, will be constructed at the mine locations to provide accommodation for owner’s team management, expatriates and consultants. Single units will comprise of one and two bed shared ablution facilities and family units with two bedrooms and bathroom with open plan living room and kitchen.

The Owners Camp will be constructed upfront and utilised as the project construction camp. The camp will accommodate the construction workers during execution and will be erected within walking distance of the operations.

Cell phone service and satellite internet are currently available on both sites as well as in Kolwezi. Fibre optic internet connectivity is foreseen in the near future.

An integrated approach to waste management for the Kamoa-Kakula Project will be required. This would involve reduction, reuse, recycling and would be done onsite through waste separation. A non-hazardous landfill site is planned at the Kamoa-Kakula Project.

24.7.6 Construction Facilities

To facilitate the execution of the project, various temporary facilities need to be put in place. These facilities include:

- Construction Site Offices: The Mine Services Building will be constructed upfront to accommodate the client site team as well as the EPCM consultants. These offices will include ablutions and conference rooms and will have facilities to communicate with head offices and receive and print construction drawings.
- Laydown areas: Contractors will require prepared areas to establish their site offices and areas to store construction material, equipment and vehicles. Fenced terrace areas with water, sewer and temporary electrical connections will be provided.
- Customs Clearance Area (Bonded Area): To facilitate the smooth delivery and release of construction material ordered from outside the DRC, a customs clearance area will be created on site from which a customs clearance official will check, register and release all imported construction material. Fenced terrace areas with office, small store, water, sewer and electrical connections will be provided.

Earthworks shall be designed with suitable grading for quick elimination of surface run-off and keeping in mind optimisation of cut-and-fill earthworks quantities. Stepped terraces shall be proposed to accommodate mechanical and process requirements on the plant.
The Kansoko site has been identified to consist of collapsible soils of low bearing capacity that shall prove inadequate to support heavy structural foundation loads such as the mills, which will therefore require piling. The Kakula site soil conditions were assumed to be similar to the Kansoko soil conditions and will require further geotechnical investigation.

24.7.7 Infrastructure Capital and Operating Costs

24.7.7.1 Infrastructure Capital Cost

Capital costs are defined as the expenditure required during the design, construction and commissioning phases of the project. This includes all costs associated with labour, construction, equipment, bulk materials, other materials, permanent equipment, sub contracts, packaging, transportation, loading, off-loading and capital indirect costs which contribute to the physical construction of the infrastructure.

Capital cost estimate for infrastructure associated with Kakula and Kansoko sites are in the section on infrastructure cost breakdown. The capital costs for Kakula and Kansoko infrastructure made allowance for the following items:

- Concentrator support infrastructure.
- Mine support infrastructure.
- Power.
- Fuel infrastructure.
- TSF.
- Accommodation infrastructure.

24.7.7.2 Infrastructure Operating Costs

The operating cost estimates for the Kakula Mine and Kansoko Mine includes fixed labour and maintenance costs and variable cost components. Operating cost estimate for infrastructure associated with Kakula and Kamoa sites are detailed in Section 24.3, operating cost tables. The operating cost figure excludes rehabilitation, insurance costs, import duties and all other taxes.

The sources of information and assumptions are as follows:

- Vendor information and quotations based on the Kamoa 2017 PFS study, factored.
- Plant labour and staffing levels as supplied by Kamoa Copper SA.
- Power cost supplied by Kamoa Copper SA.
- MDM Technical Africa (Pty) Ltd (MDM) knowledge and experience.

24.7.8 Comments on Section 24.6

During the infrastructure planning for the Kakula 2017 PEA no issues were identified that may have a material negative impact on the financial viability of the project. Synergy with regards to shared infrastructure, with possible resultant cost reductions, will be reviewed between the Kansoko and Kakula infrastructure during the next stage of the study.

24.8 Kakula 2017 PEA G&A and Owners Costs

Owners and General and Administration (G&A) costs were developed using factored fixed costs and unit rates from the Kamoa 2017 PFS.

25 INTERPRETATION AND CONCLUSIONS

25.1 Kamoa-Kakula 2018 Resource Update

The Kamoa-Kakula 2018 Resource Update provides an update of the Kamoa-Kakula Project Mineral Resource, with the Mineral Reserve from the Kamoa-Kakula 2017 Development Plan and the results of the preliminary economic assessment (PEA) from the Kamoa-Kakula 2017 PEA remaining the same. Aside from the updated Mineral Resource, further study work is currently incomplete and has not determined any results that require material changes to the Kamoa 2017 PFS or Kamoa-Kakula 2017 PEA.

Now that a Mineral Resource estimate has been independently verified for the Kakula West Discovery, Ivanhoe and Zijin can explore options to accelerate future mine production by bringing high-grade mineralization from Kakula West into the Kakula mine plan.

Additional exploration success could have a significant influence on the size, value and timing of the overall development plan; as such, the Kamoa-Kakula development plans will be reassessed and amended as the project moves forward to reflect ongoing exploration results.

25.2 Mineral Resource Estimate

Mineral Resources for the Project have been estimated using core drill data, have been performed using industry best practices (CIM, 2003), and conform to the requirements of CIM Definition Standards (2014). Amec Foster Wheeler has checked the data used to construct the resource models, the methodology used to construct it (Datamine macros), and has validated the resource models. Amec Foster Wheeler finds the Kamoa resource models to be suitable to support prefeasibility-level mine planning. The Kakula resource model is suitable to support preliminary economic assessments.

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

- Drill spacing.
  - The drill spacing at the Kamoa and Kakula deposits is insufficient to determine the effects of local faulting on lithology and grade continuity assumptions. Local faulting could disrupt the productivity of a highly-mechanised operation. In addition, the amount of contact dilution related to local undulations in the SMZ has yet to be determined for both deposits. Ivanhoe plans to study these risks with the declines currently in progress at Kamoa. A similar decline is being developed to provide access to the Kakula deposit.
  - Delineation drill programs at the Kamoa deposit will have to use a tight (approximately 50 m) spacing to define the boundaries of mosaic pieces (areas of similar stratigraphic position of SMZs) in order that mine planning can identify and deal with these discontinuities. At the Kakula deposit, the mineralisation appears more continuous compared to Kamoa.
- Assumptions used to generate the data for consideration of reasonable prospects of eventual economic extraction for the Kamoa deposit.
  - Mining recovery could be lower and dilution increased where the dip locally increases

on the flanks of the domes. The exploration decline should provide an appropriate trial of the conceptual room-and-pillar mining method on the Kamoa deposit in terms of costs, dilution, and mining recovery. The decline will also provide access to data and metallurgical samples at a bulk scale that cannot be collected at the scale of a drill sample.

- Assumptions used to generate the data for consideration of reasonable prospects of eventual economic extraction for the Kakula deposit.
  - A controlled convergence room-and-pillar technique is being studied which provides the opportunity for reduced costs.
- Metallurgical recovery assumptions at Kamoa.
  - Metallurgical testwork at the Kamoa deposit indicates the need for multiple grinding and flotation steps. Variability testwork has been conducted on only portions of the Kamoa deposit. Additional variability testing is needed to build models relating copper mineralogy to concentrate grade and improve the recovery modelling.
  - A basic model predicting copper recovery from certain supergene mineralisation types has been developed. More variability testing is required to improve this model to the point where it is useful for production planning purposes.
- Metallurgical recovery assumptions at Kakula.
  - Preliminary metallurgical testwork at the Kakula deposit indicates that a high-grade chalcocite-dominant concentrate could be produced at similar or higher recoveries compared to those achieved for Kamoa samples.
  - There is no supergene mineralisation currently identified at Kakula that requires a dedicated recovery model separate from the hypogene recovery prediction method.
- Exploitation of the Kamoa-Kakula Project requires building a greenfields project with attendant infrastructure. Changes in the assumptions as to operating and capital costs associated with the proposed development may affect the base case cut-off grades selected for the Kamoa and Kakula Mineral Resource estimates.
- Commodity prices and exchange rates.
- Cut-off grades.

25.3 Kamoa-Kakula Development Plan

The development of Kamoa-Kakula should be reassessed for the impact of the Kakula West Mineral Resource in order to determine the relative value of Kakula West against the other areas within the Kakula and Kamoa Mineral Resources.

The Kamoa-Kakula 2017 Development Plan includes an update of the Kamoa Mineral Reserve and updates of the preliminary economic assessment (PEA) on the Kakula Mineral Resource. The production rate assumption at each deposit increased from 4 Mtpa to 6 Mtpa and the total combined production rate increased from 8 Mtpa to 12 Mtpa.

The Mineral Reserves for the Kamoa 2017 PFS increased as a result of an increase in production rate through a change to the controlled convergence room-and-pillar mining method.

The analysis in the Kamoa-Kakula 2017 PEA indicates that discovery of the Kakula deposit has changed the potential development scenarios for the Kamoa-Kakula project, and additional studies should be prepared to define the development sequence and production rates including mining methods, plant sizing and location for the deposits.

The Kamoa-Kakula 2018 Resource Update is an update of the Kakula Mineral Resource including Kakula West. The development scenario should be tested and reviewed to determine updates to the development plan.

25.4 Mineral Reserve Estimate

Mineral Reserves for the Kamoa 2017 PFS conform to the requirements of CIM Definition Standards (2014). Stantec has utilised development processes and cost estimates to the level of accuracy required to state reserves and support a prefeasibility-level study.

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

- Commodity prices and exchange rates.
- Ground reaction to the controlled convergence room-and-pillar mining method. To address this, the schedule allows for a trial panel to be 80% extracted prior to beginning other controlled convergence room-and-pillar areas.
- The continuity and dip of the ore will need to be better defined prior to and during the mining stages.

25.5 Risk

The risks and uncertainties described below are not the only risks and uncertainties that the Kamoa-Kakula Project faces. Additional risks and uncertainties which have not been identified or are currently believed to be immaterial may also adversely affect the results. If any of the possible events described below occurs, the Kamoa-Kakula Project could be materially and adversely affected:

- The Kamoa-Kakula Project may not achieve its production estimates, and the development into a commercially viable mine cannot be assured.
- The Kamoa-Kakula Project requires significant infrastructure development in order to commence development and mining operations.
- Future development depends on adequate infrastructure. In particular, reliable power sources, water supply, transportation and surface facilities are key requirements that are needed to develop a mine. Any failure to address these infrastructure requirements could affect the ability to commence or continue production.
- Unusual or infrequent weather phenomena, natural disaster such as earthquake, government regulations or other interference in the provision or maintenance of such infrastructure, sabotage or terrorism, could have a material adverse effect on Ivanhoe's business, financial condition, results of operations or prospects.
- The Kamoa-Kakula Project will require additional approvals, licences and permits that it currently does not have, to commence mining operations.
- The Kamoa-Kakula Project will need substantial additional financing in the future and cannot assure that such financing will be available.
- Title to the Project cannot be assured.
- Any dispute, revocation or challenge of mineral title could have a material adverse effect.
- Legal protections in the DRC may be limited.
- Ivanhoe's operations in the DRC are subject to numerous risks associated with operating in emerging economies.
- There is a risk of direct government intervention in Ivanhoe's mineral property interests in DRC.
- The success of the Kamoa Project in meeting forecast cash flows will be largely dependent on the future price of copper.
- The expected mining extraction ratios might reduce after more detailed geotechnical studies are completed.
- The ability of Ivanhoe to attract qualified personnel in DRC may be affected by crime, poor social institutions, legal restrictions and political and economic instability.
- Currency fluctuations may affect the costs.
- Mining operations are subject to laws and regulations relating to the protection and remediation of the environment.

- The quantity and inflow of water will need to be assessed and may impact the mining plans and costs.
- As a participant in the resource extraction industry, Ivanhoe may face opposition from local and international groups.
- The costs of complying with applicable laws and governmental regulations may have an adverse impact on the business.
- Internal controls and procedures may not be sufficient to ensure compliance with its anti-bribery and anti-corruption requirements.
- Ivanhoe's insurance coverage does not cover all of its potential losses, liabilities and damages related to its business and certain risks are uninsured or uninsurable.
- Mining is inherently dangerous and subject to factors or events beyond Ivanhoe's control.
- It may not be possible to effect service of process and enforce judgments outside of Canada.
- Competition in the mining industry may adversely affect Ivanhoe.
- Ivanhoe is dependent on qualified personnel.
- Labour disruptions and/or increased labour costs could have an adverse effect on the Project.
- The Company faces certain risks in dealing with HIV/AIDS and tuberculosis.

26 RECOMMENDATIONS

26.1 Further Assessment

Ivanhoe now has three areas within the Kamoa-Kakula Project (Kamoa, Kakula and Kakula West) that warrant further assessment and are at different stages of study and development. Kakula is a very high-grade Mineral Resource that is separate to Kamoa and could be developed as a separate mine and processing facility, and given this, further study should be undertaken. The Kakula 2017 PEA has identified potential development scenarios for Kamoa and Kakula deposits.

The findings and recommendations of the Kamoa 2017 PFS remain current, and further studies on the Kamoa deposit are in progress but are not yet complete.

A whole of project approach should be undertaken to optimise the project and to take the project through the study phases to production. The next phase of study should be to prepare a PFS on Kakula. These additional studies will assist in further defining the scope for the next studies of the overall development of the entire Kamoa-Kakula Project. The key areas for further studies are:

- Commence PFS of the Kakula deposit.
- Revisions and updates of the long-term whole of project planning as the Mineral Resources are further defined. Including expanding and optimising the project production rate by considering concentrator and smelter capacities that are matched to the power supply availability, mine production and transport options.
- Other mining areas and additional mines from the Kamoa deposit.
- Rail transport to Lobito.
- Continue infill drilling programme to upgrade resource categorisation, enhance geotechnical database and its application to mine design and ground support, and better understand the continuity of the deposit and impacts on productivities and dilution.
- Consider an underground exploration programme at Kakula to attain first-hand information on actual mining conditions and to validate design assumptions.
- Complete hydrological studies and data evaluation to better determine impacts on underground mining conditions and productivities.

26.2 Drilling

An initial drill programme to complete 129,000 m at a cost of US$21.2 M that commenced in May 2016 at Kakula. As a result of the positive results this program was extended and at the end of 2017, a total of 177,860 m have been completed by both in-house drilling and contractor rigs.

Drilling is planned to continue at a similar rate in 2018. Amec Foster Wheeler has recommended a total programme of approximately 109,000 m planned at a cost of US$19.5 M. The drill targets will be defined as ongoing results become available, but expansion and infill at Kakula West remains a priority, as well as additional exploration drilling planned to test targets elsewhere within the Project.

26.3 Underground Mining

The following is a list of mining recommendations for the Project:

- Monitor the initial panel of the controlled convergence room-and-pillar mining to attain first-hand information on actual mining conditions and to validate design assumptions.
- Continue the infill drilling programme to upgrade resource categorisation, enhance geotechnical database and application to mine design and ground support, and better understand the continuity of the deposit and impacts on productivities, recoveries, and dilution.
- Consider an underground exploration programme at Kamoa to attain first-hand information on actual mining conditions and to validate design assumptions.
- Complete hydrological studies and data evaluation to better determine impacts on underground mining conditions and productivities at Kamoa.
- Drill geotechnical holes to determine ground conditions at each ventilation raise.
- Monitor KPS zones for changing ground conditions and apply the findings.
- Determine the virgin rock temperature gradient.
- Develop an operating philosophy to optimize waste rock going into room-and-pillar and goaf areas.
- Perform a detailed simulation of the underground traffic flow at peak production.
- Conduct a survey of the local workforce to determine available skill levels. The mining productivities and costs have assumed that skilled tradesmen are available to fill the critical mine operational positions.

26.4 Process Plant

The following is a list of process recommendations for the Kamoa deposit:

- Kamoa Copper SA should develop a reliable and economic measurement method to determine the copper mineralogy of samples. This will be able to predict concentrate grades and copper recoveries. Planned variability testing must proceed and the suitability of the IFS4a flotation flowsheet must be critically analysed in light of the variability results. The most critical unresolved process issue is prediction of copper concentrate grade and recovery to a level that will support production planning requirements.

- Anomalies in the current Crusher Work Index (CWI) determinations need to be resolved with additional testing of the variability samples. Subsequently, the crusher designs may require updating.
- A reliable prediction method is required for copper concentrate grade, based on either the Cu:S ratio or on measured copper mineralogy. A variability testwork program must be performed to establish, at a minimum, a useful predictive method.
- If a smelter is considered for future studies, then the concentrate grade prediction method requires a high level of accuracy when compared to a concentrate sales based project. Incorporation of a smelter in a PFS will require a more extensive characterization and flotation variability testwork program compared to a PFS that excludes smelting.
- The value of using %ASCu in determining copper recovery from surface-oxidised supergene samples must be confirmed by a program of sample analysis and flotation variability testwork.
- The current method of predicting copper recovery using %ASCu, assuming it is proven useful, should be targeted for refinement in the variability flotation testwork program.
- The currently preferred ASCu determination method may be dissolving copper that is easily floatable (chalcocite and covellite) and alternative methods (weaker acid, alternative acids, etc) should be explored within the flotation variability testwork program.

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

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