OreWin

Independent Mining Consultants

IVANHOE MINES LTD.

Kamoa 2016 Prefeasibility Study

March 2016

Job No. 15003

IMPORTANT NOTICE

This notice is an integral component of the Kamoa 2016 Prefeasibility Study (Kamoa 2016 PFS) and should be read in its entirety and must accompany every copy made of the Kamoa 2016 PFS. The Kamoa 2016 PFS has been prepared using the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects.

The Kamoa 2016 PFS has been prepared for Ivanhoe Mines Limited (Ivanhoe) by OreWin Pty Ltd (OreWin), Amec Foster Wheeler E&C Services Inc, (Amec Foster Wheeler), SRK Consulting (South Africa) Pty Ltd (SRK), and Stantec Consulting International LLC (Stantec) as the Report Contributors. The Kamoa 2016 PFS is based on information and data supplied to the Report Contributors by Ivanhoe and other parties. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the services of the Report Contributors, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. Each portion of the report is intended for use by Ivanhoe subject to the terms and conditions of its contract with the Report Contributors. Except for the purposes legislated under Canadian provincial and territorial securities law, any other uses of the report, by any third party, is at that party's sole risk.

The conclusions and estimates stated in the Kamoa 2016 PFS are to the accuracy stated in the Kamoa 2016 PFS only and rely on assumptions stated in the Kamoa 2016 PFS. The results of further work may indicate that the conclusions, estimates and assumptions in the Kamoa 2016 PFS need to be revised or reviewed.

The Report Contributors have used their experience and industry expertise to produce the estimates and approximations in the Kamoa 2016 PFS. 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 2016 PFS will be prone to fluctuations with time and changing industry circumstances.

The Kamoa 2016 PFS should be construed in light of the methodology, procedures and techniques used to prepare the Kamoa 2016 PFS. Sections or parts of the Kamoa 2016 PFS should not be read or removed from their original context.

The Kamoa 2016 PFS is intended to be used by Ivanhoe, subject to the terms and conditions of its contract with the Report Contributors. Recognizing that Ivanhoe has legal and regulatory obligations, the Report Contributors have consented to the filing of the Kamoa 2016 PFS with Canadian Securities Administrators and its System for Electronic Document Analysis and Retrieval ("SEDAR").

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Title Page

Project Name:	KAMOA COPPER PROJECT
Title:	Kamoa 2016 Prefeasibility Study
Location:	Lualaba Province. Democratic Republic of the Congo
Effective Date of Technical Report:	29 March 2016
Effective Date of Mineral Resources:	
Underground Mineral Resources:	10 December 2012
Open Pit Mineral Resources:	12 January 2013
Effective Date of Drilling Database:	1 October 2013
Effective Date of Mineral Reserves:	29 March 2016

Qualified Persons:

- Bernard Peters, B. Eng. (Mining), FAusIMM (201743), employed by OreWin as Technical Director - Mining was responsible for: Sections 1.1, 1.2, 1.3, 1.4, 1.13, 1.14, 1.17.1, 1.17.3, 1.18.1; Section 2; Section 3; Section 4, Section 5; Section 10.7, Section 15; Sections 16.2, 16.3, 16.4, 16.5, 16.6; Section 19; Section 20; Sections 21.1, 21.2, 21.8; Section 22; Section 23; Section 24; Sections 25.2, 25.3, 26.1, 26.4; Section 27.
- Dr Harry Parker, SME Registered Member (2460450), Technical Director, Amec Foster Wheeler was responsible for: Sections 1.5 to 1.9, 1.11, 1.12, 1.17.2, 1.18.2; Section 2; Section 3; Section 6; Section 7; Section 8; Section 9; Sections 10.1 to 10.6, 10.9 to 10.11; Sections 11.1, 11.2, 11.4 to 11.11; Section 12, Section 14; Section 25.1, Sections 26.2, 26.3; Section 27.
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Signature Page

Project Name:	KAMOA COPPER PROJECT
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Location:	Lualaba Province. Democratic Republic of the Congo
Effective Date of Technical Report:	29 March 2016

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1 SUMMARY

1.1 Introduction

The Kamoa 2016 Prefeasibility Study (Kamoa 2016 PFS) is a prefeasibility study (PFS) prepared for Ivanhoe Mines Ltd. (Ivanhoe or the Company), to provide an independent Technical Report (the Report) for the Kamoa Copper Project (Kamoa Project) located in the Democratic Republic of Congo (DRC).

The Kamoa Project is a newly discovered, very large, stratiform copper deposit with adjacent prospective exploration areas within the Central African Copperbelt, located approximately 25 km west of the town of Kolwezi and about 270 km west of the provincial capital of Lubumbashi.

On 8 December 2015, Ivanhoe and Zijin Mining Group Co., Ltd. (Zijin) closed an agreement to co-develop the Project. Under terms of agreements, Zijin through its subsidiary, Gold Mountains (H.K.) International Mining Company Limited owns 49.5% share interest in Kamoa Holding Limited (Kamoa Holding). Crystal River Global Limited is a private company that owns 1% of Kamoa Holding. Kamoa Holding presently owns 95% of the Kamoa Project. The relationship between Ivanhoe Mines, Zijin, and Crystal River Global Limited will be governed by a Shareholder, Governance and Option Agreement (SGOA). The SGOA provides, among other things, that all key decisions regarding the development and operation of the Kamoa Project will be made by Kamoa Holding's Board of Directors. A 5%, non-dilutable interest in Kamoa Holding was transferred to the government of the DRC on 11 September 2012, for no consideration, pursuant to the DRC Mining Code. The company also has offered to sell an additional 15% interest to the government of the DRC on commercial terms to be negotiated.

The previous Technical Report was the Kamoa 2013 PEA which examined the construction and operation of a long-term underground mine, concentrator processing facility and smelter operation and associated infrastructure and included a production expansion.

The Kamoa 2016 PFS reflects the initial phase of project development and describes the construction and operation of a 3 Mtpa underground mine, concentrator processing facility and associated infrastructure. The Kamoa 2016 PFS includes an economic analysis that is based on Probable Mineral Reserves. The production scenario schedules 71.9 million tonnes at an average grade of 3.86% copper over 24 years, producing 6.1 Mt of copper concentrate, containing 2, 394 kt of copper in concentrate. The economic analysis used a long term price assumption of US$3.00/lb of copper. The economic analysis returns an after-tax net present value (NPV) at an 8% discount rate of US$986 million. It has an after-tax internal rate of return (IRR) of 17.2% and a payback period of 4.6 years. The initial capital cost, including contingency, is US$1.2 billion. The life-of-mine average total cash cost after credit is US$1.48/lb of copper.

Improvements to the mining method have the potential to reduce average mine site cash cost during the first phase to US$0.61/lb of copper, and improve the after-tax NPV at an 8% discount rate to US$1.182 billion, the IRR to 18.9% and the payback period to 4.3 years.

The results of the Kamoa 2016 PFS 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 2016 PFS. 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.

1.2 Kamoa 2016 PFS Results

The base case described in the study is the construction and operation of an underground mine, concentrator processing facility, and associated infrastructure. The base case mining rate and concentrator feed capacity is 3 Mtpa. The life-of-mine production scenario schedules 71.9 million tonnes at an average grade of 3.86% copper over 24 years, producing 6.1 million tonnes of copper concentrate, containing approximately 5.3 billion pounds of copper.

The economic analysis used a long term price assumption of US$3.00/lb for copper and returns an after-tax NPV at an 8% discount rate of US$986 million. It has an after-tax IRR of 17.2% and a payback period of 4.6 years.

The initial capital cost, including contingency, is US$1.2 billion. The initial capital cost includes a US$104 million advance payment to the DRC state-owned electricity company, SNEL, to upgrade two hydro power plants (Koni and Mwadingusha) to provide Kamoa with access to clean electricity during the initial phase of operations. The upgrading work is being led by Stucky Ltd., of Switzerland, and the advance payment is expected to be recovered through a reduction in the power tariff once Kamoa is in operation. The life-of-mine average mine site cash cost is US$0.75/lb of copper. The key results of the PFS are summarised in Table 1.1.

Table 1.1 Kamoa 2016 PFS Results Summary

Item	Unit	Total
Ore Processed		
Quantity Ore Treated	kt	71,893
Copper Feed Grade	%	3.86
Concentrate Produced		
Copper Concentrate Produced	kt (dry)	6,106
Copper Recovery	%	86.36
Copper Concentrate Grade	%	39.20
Contained Metal in Concentrate	Mlb	5,277
Contained Metal in Concentrate	kt	2,394
Key Financial Results		
Initial Capital	US$M	1,213
Mine Site Cash Cost	US$/lb Payable Cu	0.75
Total Cash Costs	US$/lb Payable Cu	1.48
Site Operating Costs	US$/t ore	53.22
After-Tax $NPV_{8\%}$	US$M	986
After-Tax IRR	%	17.2
Project Payback Period	Years	4.6

Table 1.2 summarises the financial results, whilst Table 1.3 summarises mine production, processing, concentrate, and metal production statistics.

Table 1.2 Financial Results

		Before Taxation	After Taxation
Net Present Value (US$M)	Undiscounted	5,791	4,096
	4.0%	2,979	2,036
	6.0%	2,152	1,429
	8.0%	1,549	986
	10.0%	1,104	657
	12.0%	768	409
IRR	–	20.7%	17.2%
Project Payback (years)	–	4.1	4.6

Table 1.3 Mining Production and Processing Statistics

Item	Unit	Total	Years 1–5 Annual Average	Years 6–10 Annual Average	Total Annual Average
Ore Processed					
Quantity Ore Treated	kt	71,893	2,934	3,008	2,996
Copper Feed grade	%	3.86	4.35	4.08	3.86
Concentrate Produced					
Copper Concentrate Produced	kt	6,106	283	271	254
Copper Recovery	%	86.36	87.06	86.68	86.36
Copper Concentrate Grade	%	39.20	39.20	39.20	39.20
Contained Metal in Concentrate					
Copper	kt	2,394	111	106	100
Copper	Mlb	5,277	245	234	220
Payable Metal					
Copper	kt	2,314	107	103	96
Copper	Mlb	5,102	237	227	213

Figure 1.1 and Figure 1.2 depict the processing, concentrate and metal production, respectively. Table 1.4 summarises unit operating costs and Table 1.5 provides a breakdown of operating costs and revenue. The capital costs for the project are detailed in Table 1.6.

Figure 1.1 Processing Production

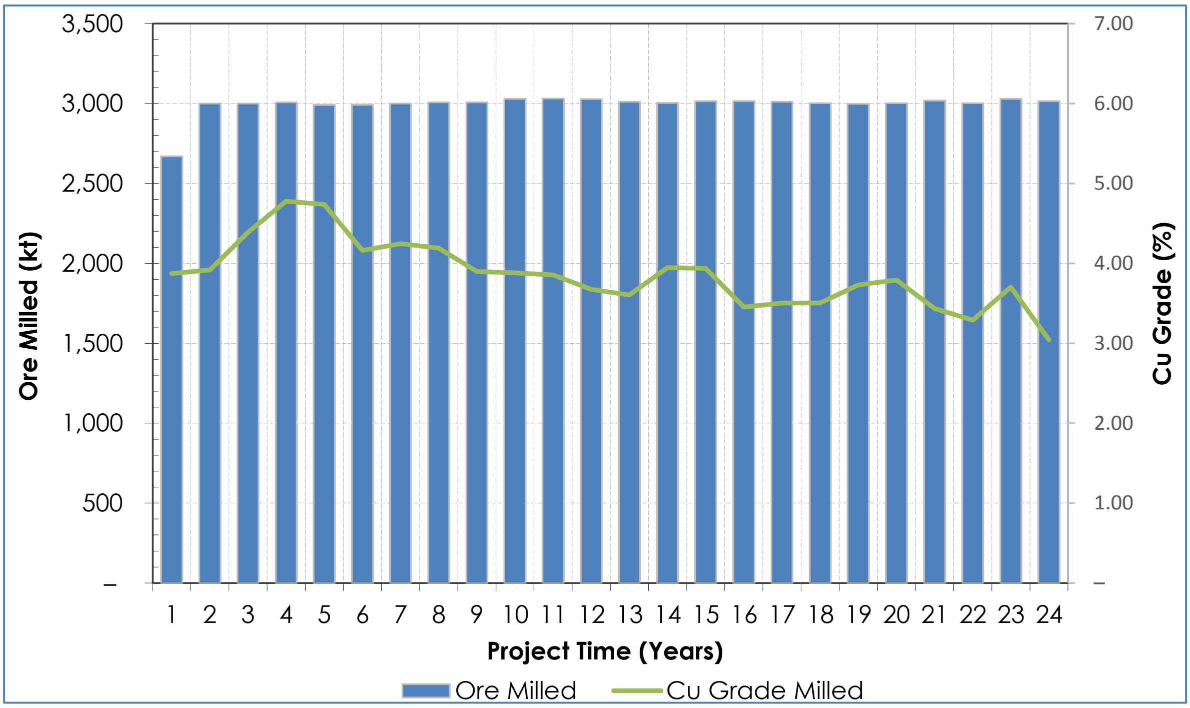


Figure by OreWin, 2016.

Figure 1.2 Concentrate and Metal Production

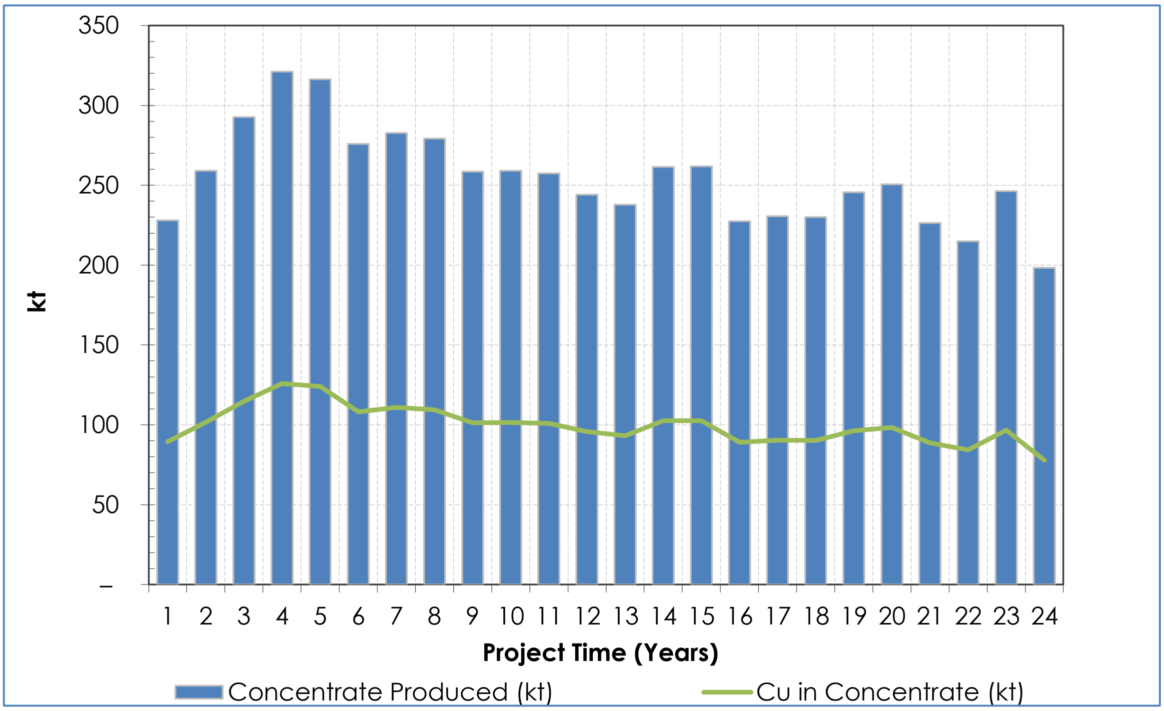


Figure by OreWin, 2016.

Table 1.4 Unit Operating Costs

	US$/lb Contained Cu in Concentrate		
	Total Average	Years 1–5 Average	Years 6–10 Average
Mine Site	0.75	0.55	0.75
Transport	0.41	0.43	0.40
Treatment & Refining Charges	0.18	0.18	0.18
Royalties & Export Tax	0.15	0.11	0.16
Total C1 Cash Costs	**1.48**	**1.27**	**1.48**

Table 1.5 Revenue and Operating Costs

Description	Total	Years 1–5 Average	Years 6–10 Average	Total Average
	US$M	US$/t Ore Milled		
Revenue				
Copper in Concentrate	15,305	242	226	213
Realisation Costs				
Transport	2,082	35	30	29
Treatment & Refining	911	14	13	13
Royalties & Export Tax	748	9	12	10
Total Realisation Costs	**3,741**	**58**	**56**	**52**
Net Sales Revenue	**11,565**	**184**	**170**	**161**
Site Operating Costs				
Underground Mining	2,453	24.6	37.1	34.1
Processing	886	12.4	12.3	12.3
Tailings	25	0.4	0.4	0.4
General & Administration	511	8.1	7.1	7.1
SNEL Discount	-109	-1.5	-1.5	-1.5
Customs	60	0.7	0.9	0.8
Total Site Operating Costs	**3,826**	**44.6**	**56.2**	**53.2**
Operating Margin	**7,738**	**140**	**114**	**108**
Operating Margin	**66.9%**	**75.8%**	**67.0%**	**66.9%**

Table 1.6 Capital Investment Summary

Description	Initial Capital	Sustaining	Total
	US$M	US$M	US$M
Mining			
Underground Mining	508	499	1,007
Subtotal	**508**	**499**	**1,007**
Power			
Power Infrastructure	19	4	23
Power Supply Off Site	104	–	104
Subtotal	**123**	**4**	**127**
Concentrate & Tailings			
Process Plant	160	8	167
Tailings	23	62	84
Subtotal	**182**	**69**	**252**
Infrastructure			
Plant Infrastructure	18	4	23
Other Infrastructure	8	2	10
Owners Camp	10	2	12
Contractors Camp	23	5	28
Subtotal	**59**	**14**	**73**
Indirects			
EPCM	58	–	58
Subtotal	**58**	**–**	**58**
Owners Cost			
Owners Cost	95	–	95
Closure	–	67	67
Subtotal	**95**	**67**	**162**
Capital Expenditure Before Contingency	**1,025**	**655**	**1,679**
Contingency	189	44	233
Capital Expenditure After Contingency	**1,213**	**699**	**1,912**

The cash flow sensitivity to metal price variation is shown in Table 1.7, for copper prices from US$2.00/lb copper to US$4.00/lb.

The sensitivity of after-tax NPV to initial capital cost, direct operating costs, transport and Cu feed grade is shown in Table 1.8.

The table shows the impact on the base case after-tax $NPV_{8\%}$ of US$986 M. The table shows the change in the base case After Tax NPV8 of US$986 M. 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. The sensitivity to transport applies the costs via Durban (US$356/t) and via Lobito (US$300/t).

Table 1.7 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	613	2,364	4,096	5,828	7,560
4.0%	-20	1,020	2,036	3,050	4,063
6.0%	-206	624	1,429	2,230	3,030
8.0%	-340	336	**986**	1,632	2,276
10.0%	-438	123	657	1,187	1,714
12.0%	-508	-36	409	851	1,289
15.0%	-579	-206	142	486	827
IRR	3.8%	11.5%	17.2%	22.2%	26.6%

Table 1.8 Additional Sensitivities

			Change from Base $NPV_{8\%}$ (US$M)				
Variable	**Units**	**Base Value**	**-25.0%**	**-10.0%**	**–**	**10.0%**	**25.0%**
Initial Capital	US$M	1,213	1,252	1,092	986	879	720
Direct operating costs per tonne of ore milled	US$/t	53	1,215	1,077	986	894	757
Transport costs per tonne of concentrate	US$/t	356 / 300	1,122	1,040	986	931	849
Copper Feed Grade	% Cu	3.86%	175	662	986	1,308	1,794

Figure 1.3 Cumulative Cash Flow

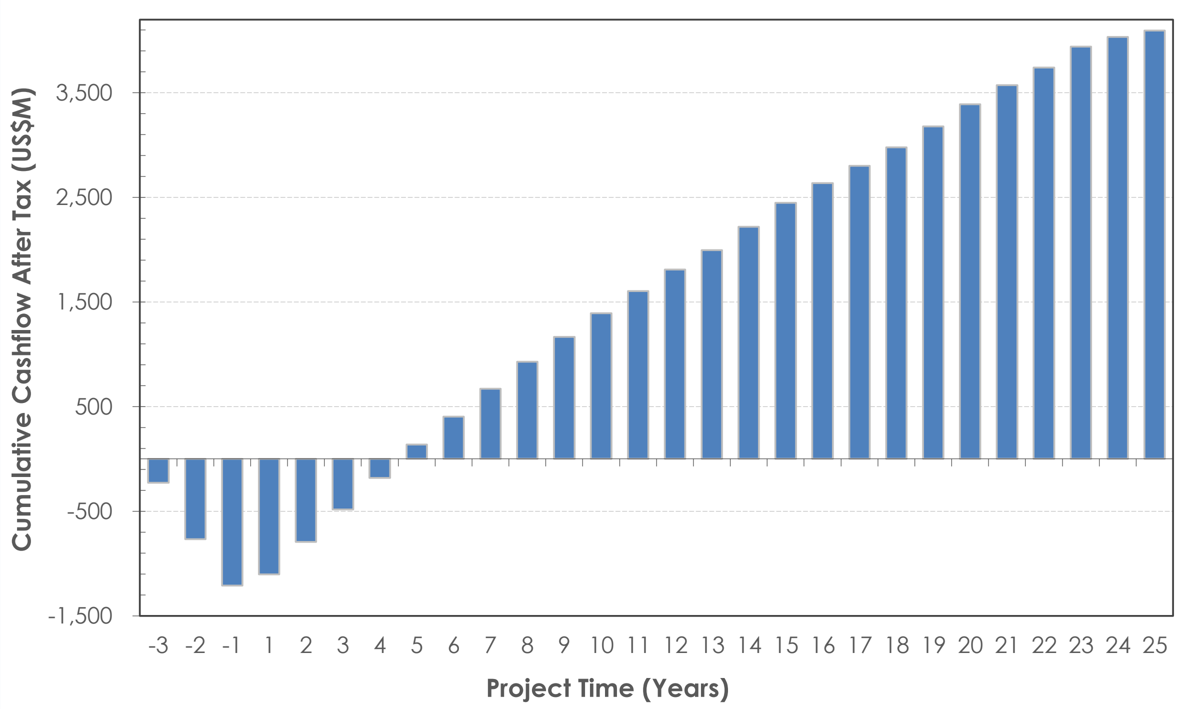


Figure by OreWin, 2016.

The concentrate copper equivalent production for a set of other copper projects that are currently under construction is compared with the Kamoa 2016 PFS in Figure 1.4. The capital intensity for the copper projects and the Kamoa 2016 PFS are compared in Figure 1.5. The two figures are based on data from Wood Mackenzie, February 2016. The Wood Mackenzie research identified the projects as greenfield development projects and was based on public disclosure and information gathered by Wood Mackenzie.

Figure 1.4 Copper Projects Under Construction Concentrate Equivalent Production

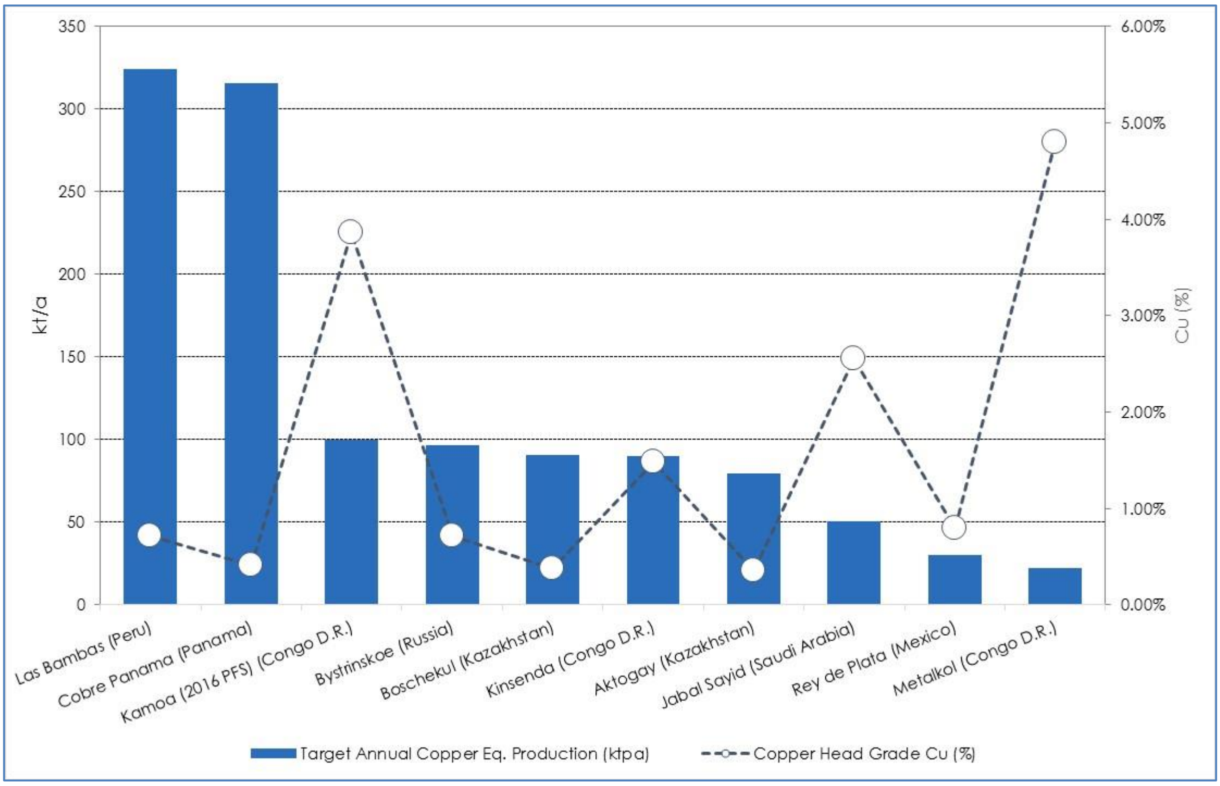


Figure based on data from Wood Mackenzie, February 2016.

Figure 1.5 Copper Projects Under Construction Capital Intensity

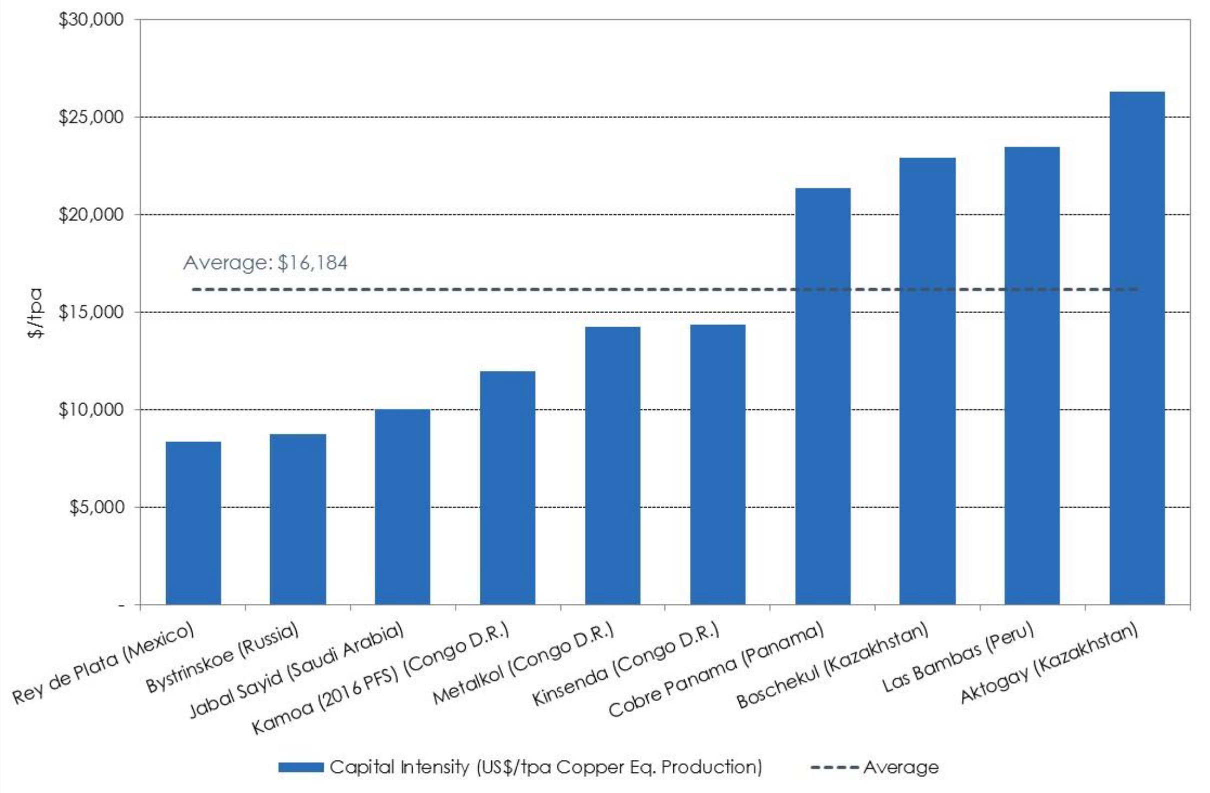


Figure based on data from Wood Mackenzie, February 2016.

1.3 Property Description and Location

The Kamoa Project is situated in the Kolwezi District of Lualaba Province, DRC. The Kamoa Project is located approximately 25 km west of the town of Kolwezi, and about 270 km west of the provincial capital of Lubumbashi. The Kamoa copper deposit was discovered by Ivanhoe Mines in 2008.

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 to the north of the Project area, 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 successfully conducted exploration programmes on a year-round basis in 2009 and again in 2011.

The Project is currently isolated from public infrastructure. Infrastructure on-site is currently limited to support for the exploration activities. Ivanhoe has established a regional exploration base camp in Kolwezi because of the availability of power and communications in the town. Exploitation of the Kamoa deposit will require building a greenfields project with attendant infrastructure.

1.4 Mineral and Surface Rights, Royalities, and Agreements

The Kamoa 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 license (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 Kamoa Project resides with Kamoa Copper, a subsidiary of Kamoa Holding, which is the holder of the Kamoa Exploitation Licences.

Ivanhoe owns a 49.5% share interest in Kamoa Holding Limited (Kamoa Holding), an Ivanhoe subsidiary that presently owns 95% of the Kamoa Copper Project. Zijin Mining Group Co., Ltd. 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.
On 11 September 2012, Ivanhoe transferred a 5% non-dilutable interest in the share capital of AMBL to a DRC state-owned nominee. Ivanhoe has offered to sell the DRC Government an additional 15% interest in the Project on commercial terms to be negotiated.

At the effective date of this Report, Ivanhoe holds no surface rights in the Project area. A single surface rights holder has been identified with formal surface rights in the project area. Negotiations are currently underway to finalise transfer of these rights to Ivanhoe.

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 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 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.

In February 2013, a draft law on the revision of the 2002 Mining Code was circulated by the DRC Minister of Mines. In February 2016 the DRC Minister of Mines announce that the the current code will be retained.

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 Kamoa, diamictites are situated in the Lower Kundelungu at its contact with Roan sandstones.

The mineralised 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). 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 ($CuFeS_2$).

The lowermost clast-rich diamictite (Ki1.1.1.1) unit generally hosts lower-grade (<0.5% Cu) 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. Copper mineralisation comprises three distinct styles: supergene, hypogene, and mixed mineralisation.

Near the surface adjacent to the Kamoa Dome, the diamictites have been leached, resulting in localized zones of copper oxides and secondary copper sulphide enrichment down-dip. Although there are several kilometres along strike of the leached boundary that present a target for this type of mineralisation, which has been encountered to depths of 400 m in drillholes collared near faults, analogue deposits in Zambia such as Mufulira East, Nkana, and Chibuluma South had relatively laterally narrow zones of oxides/supergene enrichment near surface. Hypogene mineralisation occurs at depths as shallow as 30 m.

1.6 History and Exploration

Although exploration was undertaken by the Tenke Fungurume Consortium between 1971 to 1975, and some 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.

Recent 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, and a PEA in 2013 (the Kamoa 2013 PEA) and its subsequent update (the Kamoa 2016 PFS).

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 deposit. 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 resource estimation was closed on 5 May 2014. Aircore, reverse circulation (RC) and drill 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. Core holes have been used for geological modelling and those occurring within the mining lease and in areas of mineralization (drillholes on the Kamoa and Makalu domes are excluded) have been used for resource estimation.

As at 5 November 2015, there were 1,188 core holes drilled within the broader project area. The 2014 Mineral Resource estimate used 720 drillhole intercepts. Included in the 720 drillholes are 16 twin holes (where the spacing between drillholes is <25 m) and six wedge holes. Although additional wedge holes have been drilled, the wedges have been primarily 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 in the resource estimation. The 468 holes not included in the May 2014 Mineral Resource estimate were excluded because they were either abandoned, unmineralized holes in the dome areas, unsampled metallurgical, civil geotechnical or hydrological drillholes, or have been drilled after the closure of the database. The database contains 137 drillholes that were drilled in 2014 and 2015 that were not used in the 2014 resource estimate.

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. 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 by the Amec Foster Wheeler QPs documented the core recovery to be excellent. From 2010 through 2015, all completed holes were surveyed by an independent professional surveyor, SD Geomatique, using a differential GPS that had a manufacturer accuracy claim of +/-20 mm. As of 5 November 2015, 29 drillhole collars remained to be surveyed. Downhole surveys for most drillholes were performed by the drilling contractor at approximately 30 m intervals for 2009 drilling and at a maximum of 50 m intervals for 2010 through 2015 drillholes, using a Single Shot digital downhole instrument. Multishot surveys were conducted after the holes were completed.

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

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 is convenient) above the first presence of mineralisation and/or the mineralised zone is sampled on nominal whole 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 are 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 could potentially support a mineral resource estimate. This is highlighted as "Zone of Assay" (ZA) and is extended to 3m 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 Chemex of Vancouver, British Columbia, acted as the check laboratory for drill core samples from part of the 2009 program and for 2010 through 2014 drilling. ALS Chemex is ISO:9001:2008 registered and ISO:17025-accredited.

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. 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, S, and Ag. A suite of additional elements have been requested, in particular in the early drilling phases. Acid-soluble copper (ASCu) assays have been undertaken since 2010. Very few (249 out of 6,640) samples from holes drilled prior to 2010 have ASCu assays.

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

1.9 Data Verification

Amec Foster Wheeler reviewed the sample chain of custody, quality assurance and control 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 2015, and has found no material issues with the data as a result of these audits.

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.

Bench-scale metallurgical flotation testwork carried out at XPS Consulting and Testwork Services laboratories in Falconbridge, Ontario, Canada, has shown positive results. The most recent work was conducted on composite samples of drill core from the Kansoko Sud and Kansoko Centrale areas in the southern part of the Kamoa Mineral Resource area.
Two master composite samples were formulated; one representative of Years 1 to 4 of planned mine production, and the second representative of Years 5 to 15 of mine production. In addition, a blend of 90% hypogene and 10% supergene representing production years 1 to 4 was prepared. Testwork on the 90:10 composite grading 3.61% copper, produced a copper recovery of 85.4% at a concentrate grade of 37.0% copper. Material from the later years of mining, and grading 3.20% copper, produced a copper recovery of 89.2% at a concentrate grade of 35.0% copper using the same flowsheet.

Average arsenic levels in the concentrate are approximately 0.02%, which is significantly lower than the rejection limit of 0.5% imposed by Chinese smelters and much lower than the typical penalty level of 0.2%. Very low arsenic levels in concentrate are expected to attract a premium from copper-concentrate traders.

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 millimetres (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 10µm 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 then is bagged for shipment to market.

1.11 Mineral Resource Estimates

The Mineral Resources are estimated based on drilling up to 5 May 2014, and are supported by 720 mostly vertical drillholes. Areas outlined by core drilling at 800 m spacing with a maximum extrapolation distance of 600 m between drill sections, and which show continuity of grade at 1% Cu, geological continuity, and continuity of structure (broad anticline with local discontinuities that are likely faults) were classified as Inferred Mineral Resources over an area of ~16.8 km^2. Mineral Resources within an area of 50.5 km^2 drilled on 400 m spacing and which display grade and geological continuity were classified as Indicated Mineral Resources. The total area of the Kamoa Project is 410.14 km^2.

The best single mineralised intercept (SMZ) in the appropriate stratigraphic position for each of the 720 holes within the resource boundary was selected using the criteria of a minimum copper grade of 1% Cu, and a minimum downhole length of three metres. In the event that the assays in a drillhole could not be combined to meet the above criteria, the highest-grade composite was formed that met the 3 m minimum length.

The Mineral Resource area was divided into nine structural domains, and a digital terrain model (dtm) was constructed through the SMZ centroids to define the geometry of the mineralisation within each structural domain.

A prototype gridded-seam block model was established using 100 m x 100 m blocks in the X and Y directions and a single block in the vertical direction. The Z value of the block centroid was set to the vertical midpoint of the SMZ surface using the SMZ dtm. True thickness, total copper weighted by vertical thickness and acid-soluble copper were estimated into the block model using inverse distance to the second power.

The vertical height of the resource model blocks was set to the estimated vertical thickness of the mineralisation.

Mineral Resources were classified using a nominal 400 m drillhole spacing for classification of Indicated and a nominal 800 m spacing for Inferred.

Amec Foster Wheeler used a 1% Cu cut-off grade as a base case to declare Mineral Resources. 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 a natural cut-off. The 1% Cu cut-off is also a "natural" cut-off for the Kamoa deposit, 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 based on the metallurgical recovery algorithms used in the Kamoa 2013 PEA and Kamoa 2016 PFS as appropriate. Operating and realization costs were sourced from the Kamoa 2016 PFS.

1.11.1 Mineral Resource Statement

The Mineral Resources have been defined taking into account the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves. Dr. Harry Parker, SME Registered Member, and Gordon Seibel, SME Registered Member, both employees of Amec Foster Wheeler, are the Qualified Persons for the Mineral Resource estimates. The Mineral Resources have an effective date of 5 May 2014.

Table 1.9 Indicated and Inferred Mineral Resources

Category	Tonnage (Mt)	Area (km²)	Copper (%)	True Thickness (m)	Contained Copper (kt)	Contained Copper (billions lbs)
Indicated	752	50.5	2.67	5.24	20,110	44.3
Inferred	185	16.8	2.08	3.87	3,840	8.5

1. Dr. Harry Parker and Gordon Seibel, both RM of SME, employees of Amec Foster Wheeler, are the Qualified Persons for the Mineral Resource estimate. The effective date of the estimate is 5 May 2014. Mineral Resources are reported inclusive of Mineral Reserves.
2. Mineral Resources are reported using a total copper (TCu) cut-off grade of 1% TCu and an approximate minimum true thickness of 3 metres. There are reasonable prospects for eventual economic extraction under assumptions of a copper price of US$3.30/lb; employment of underground mechanized 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 $34/t. Concentrator and General and Administrative costs are assumed to be $19/t. Metallurgical recovery will be 77% (supergene) and 85% (hypogene) at the average grade of the resource.
3. Reported Mineral Resources contain no allowances for hangingwall or footwall contact boundary loss and dilution. No mining recovery has been applied.
4. For Indicated Mineral Resources, 97.4% of the resource model blocks have a true thickness greater than 3 metres (range from 2.3 m to 15.8 m), for Inferred Mineral Resources, 94.7% of the resource blocks have a true thickness greater than 3 m (range from 2.7 m to 8.4 m).
5. Depth of mineralization below the surface ranges from 10 metres to 1,320 metres for Indicated Mineral Resources and 20 metres to 1,560 metres for Inferred Mineral Resources.
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.

A review of the block model was undertaken, applying higher cut-offs in defining the SMZs to allow for high-grade, narrower, mining options to be tested in support of potentially improving initial mine economics. Following a review of the various cut-offs, it was decided to proceed with mine planning using a 1.5% TCu cut-off grade. The minimum thickness was 3.0 m. The maximum thickness was 6.0 m; this was relaxed if material above 2.0% TCu was encountered above the 6.0 m limit.

1.11.2 Factors Which May Affect the Resource Estimates

Factors which may affect the Mineral Resource estimate include:

- Commodity prices and exchange rates.
- Cut-off grades.
- Metallurgical recoveries.
  - Metallurgical testwork indicates the need for multiple grinding and flotation steps. Metallurgical variability testwork has been initiated and covered has only is in early stages and has only covered a portion of the deposit. Recent testwork has shown improved recoveries and concentrate grades compared to previous work.
- Mining plan.
  - The presence of 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. Mitigation plans for these risks could include an exploratory decline, or, potentially, a program of inclined drillholes.
  - Delineation drill programs 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.
  - Mining recovery could be lower and dilution increased where the dip locally increases on the flanks of the domes. An exploration decline is being driven which would provide an appropriate trial of the conceptual room-and-pillar mining method on the Kamoa deposit in terms of costs, dilution, and mining recovery. A decline would provide access to data and metallurgical samples at a bulk scale that cannot be collected at the scale of a drill sample. The extent of the decline and associated drifts are dependent on further studies.
- Infrastructure.
  - Exploitation will require building a greenfields project with attendant infrastructure.
- Capital and operating costs.
  - Exploitation will require building a greenfields project with attendant infrastructure.

1.12 Targets for Further Exploration

Two targets for further exploration (referred to as exploration targets for the purposes of this report) have been identified. They are referred to in this subsection as the Kamoa–Makalu exploration target, and the Kakula Discovery exploration target.

The area inside the model perimeter surrounding the Indicated and Inferred Mineral Resources at Kamoa–Makalu is considered an exploration target. The ranges of the exploration target tonnages and grades are summarised in Table 1.10. Tonnage and grade ranges were estimated using an inverse distance weighting and applying a +/-20% variance to the resulting tonnage and grade estimate.

The Kakula Discovery exploration target within the Kakula Exploration area is based on the results obtained in eight drillholes, five of which show thick intercepts of high-grade mineralization. A success-rate of 80% was applied to reflect uncertainty as to the position of thin or leached mineralization bounding the target. The range of tonnages and grades was then derived by applying a variance factor of +/-20%.

Amec Foster Wheeler cautions that the potential quantity and grade of the exploration targets are conceptual in nature, and that it is uncertain if additional drilling will result in the exploration targets being delineated as a Mineral Resource.

Table 1.10 Tonnage and Grade Ranges for Exploration Targets

Target	Low-range Tonnage Mt	High-range Tonnage Mt	Low-range Grade (%Cu)	High-range Grade (%Cu)
Kamoa-Makalu	480	720	1.5	2.3
Kakula Discovery	580	870	1.5	2.3

The planned Kakula exploration drilling program consists of 19,000 metres in 58 holes, of which approximately 10,000 metres of drilling is planned on an 800-metre infill grid at the Kakula Discovery area. This drilling is intended to support an initial Mineral Resource estimation, and the drill spacing is approximately that required for Inferred Mineral Resources at Kamoa. The remaining approximately 9,000 metres of drilling will be used to test the other two areas of exploration potential within the Kakula exploration area.

1.13 Mineral Reserve

The Kamoa 2016 PFS Mineral Reserve has been estimated by Qualified Person Bernard Peters, Technical Director – Mining, OreWin Pty Ltd. using the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves to conform to the Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects. The total Mineral Reserves for the Kamoa project are shown in Table 1.11. The Mineral Reserve is based on the May 2014 Mineral Resource reported in the Kamoa 2016 PFS. 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 18 February 2016.

The production plan defined for the Kamoa 2016 PFS Mineral Reserve represents the first phase of the development plan defined in the Kamoa 2013 PEA. The Kamoa 2013 PEA production scenario assumed that there would be an initial period of concentrate production, followed by an expansion of the mine and concentrator and construction of a smelter.

Table 1.11 Kamoa 2016 PFS Mineral Reserve

	Ore (Mt)	Cu (%)	Recovered Cu	
			(Mlb)	(kt)
Proven Mineral Reserve	–	–	–	
Probable Mineral Reserve	71.9	3.86	5,102	2,314
Mineral Reserve	71.9	3.85	5,102	2,314

1. The copper price used for calculating the financial analysis is a long term copper at $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.
2. For mine planning the copper price used to calculate block model Net Smelter Returns was $3.00/lb.
3. An elevated cut-off grade of $100.00/t NSR was used to define the stoping blocks. A marginal cut-off grade of 1% Cu was used to defined ore and waste.
4. Indicated Mineral Resources were used to report Probable Mineral Reserves.
5. The reference point for the Mineral Reserves is mill feed.
6. The Mineral Reserves reported above are not additive to the Mineral Resources.

1.14 Mining

1.14.1 Geotechnical

SRK have completed an analysis of all provided drillhole and laboratory data, and undertaken a litho-domain based rock mass characterization for the Kamoa project. The drillhole data and all provided surface GIS and geophysics images were used by SRK to revise the 3D structural geology fault network model across the project.

The provided resource model was utilised to investigate key geometry aspects of the ore body that would influence the recommended mining methods, pillar layouts and resulting extraction ratios.

1.14.1.1 Pillar Design

The pillar loads were determined using the Tributary Area Theory, The pillar strengths and extraction percentages are based on the summarized data obtained from drill core only. The information is considered representative but needs to be verified through data collection from underground exposures.

Joint orientations are based on oriented core data, and the design assumes all pillars are affected by identical structure. Weathering has been included to account for the oxidation and rock mass degradation observed to occur in drillcore.

It is recommended, at this stage of the knowledge and uncertainty about the geological and geotechnical nature of the deposit, that the pillar FoS should be in the range of 1.4 to 1.6 to ensure stable pillars. Pillars along tramming routes and long term excavation should be designed with a FoS > 2.

The pillar and excavation dimensions provided are the final dimensions. The design does not cater for additional overbreak through poor blasting or kinematic failure.

40 m wide barrier pillars that straddle the major domain boundary faults may not be required or reduced in width once these faults are exposed underground and the width of the damage zones are investigated further.

1.14.1.2 Ore Body Geometry

SRK has utilized GIS to assist in spatially assessing the potential variability in ore body dip, thickness, and lithological boundaries, for the most recent provided resource. Kansoko Sud mining area is heavily impacted by ore body dip, with approximately 64% of the area having a dip greater than 12.6° (maximum true dip to use conventional Room and Pillar layout). Where the dip exceeds 12.6° a stepped room and pillar layout will be required. The extraction ratio will remain the same as for a conventional Room and Pillar layout, however recovery will decrease as a wedge of waste will be mined from the floor. Alternatively a Drift and Fill method can be used with cemented tails backfill (~5% cement) in steeply dipping areas.

The Local ore body geometry is not yet adequately defined. Parts of the ore body may be complexly faulted with displacements of only a few metres. This needs to be investigated with future exposure of the rock, and may impact the nature of the chosen mining method.

The nature and variability of the contacts of the ore body are also not clearly understood by SRK at this stage. Dilution factors used in the reserve will depend on the ability of the geologists to determine the location of ore contacts. This may also impact the nature of the chosen mining method.

1.14.1.3 Structure and Groundwater

The current structural model defines different categories of structures depending on the level of confidence in their existence and their expected continuity. Early stage rock exposure will help define the local continuity and geometry of the exposed fault systems, which will help characterize their impact on the rock mass conditions and groundwater permeability.

SRK recommends that the 20 m wide barrier pillars are left on either side of the primary Domain Boundary Faults on the maps in this report (provided to Ivanplats as the faults that could influence the resource). At this phase of the study, it is assumed that the other faults have limited damage zones, local continuity and displacements, and can therefore be managed with local increased rock support during mining.

1.14.1.4 Recommendations for Future Work

- For the next stage of study the current ground conditions and structural geology assumptions must be must verified from underground exposures. SRK recommends to undertake comprehensive structural and geotechnical mapping and geotechnical instrumentation programme in the proposed trial mining area.
- A more focused assessment of hanging wall conditions and its implications support should be carried out.

- A suitable program of stress measurements should be undertaken when underground access becomes available.
- Additional laboratory rock testing should be undertaken to obtain a good understanding of the range of strength across types and the distribution of lower and higher strength through all the domains. Testing should also be directed at understanding strength anisotropy as well.
- The groundwater model for the current PFS study needs to include the current defined fault network. Different groundwater transmissivities can be assumed for each type of defined structures, range from greatest transmissivities in the Domain Boundary Faults, to least transmissivities in the very low confidence fault/fracture systems.

1.14.2 Kamoa 2016 PFS Mining

The Kamoa 2016 PFS Mineral Reserve ranges between depths of 170 m and 1,200 m below surface and the average dip is approximately 16 degrees. Given the favourable mining characteristics of the Kamoa Mineral Resource, it is considered amenable to large-scale, mechanized, room-and-pillar or drift-and-fill (D&F) mining. The dip and geometry of the resource make it conducive to stepped room-and-pillar (SR&P) mining in the shallow portions of the deposit, transitioning to drift-and-fill mining in the deeper sections. The arrangement of the declines at Kansoko Sud and the Mineral Reserve mining areas are shown in Figure 1.6.

Figure 1.6 Mineral Reserve Mining Areas

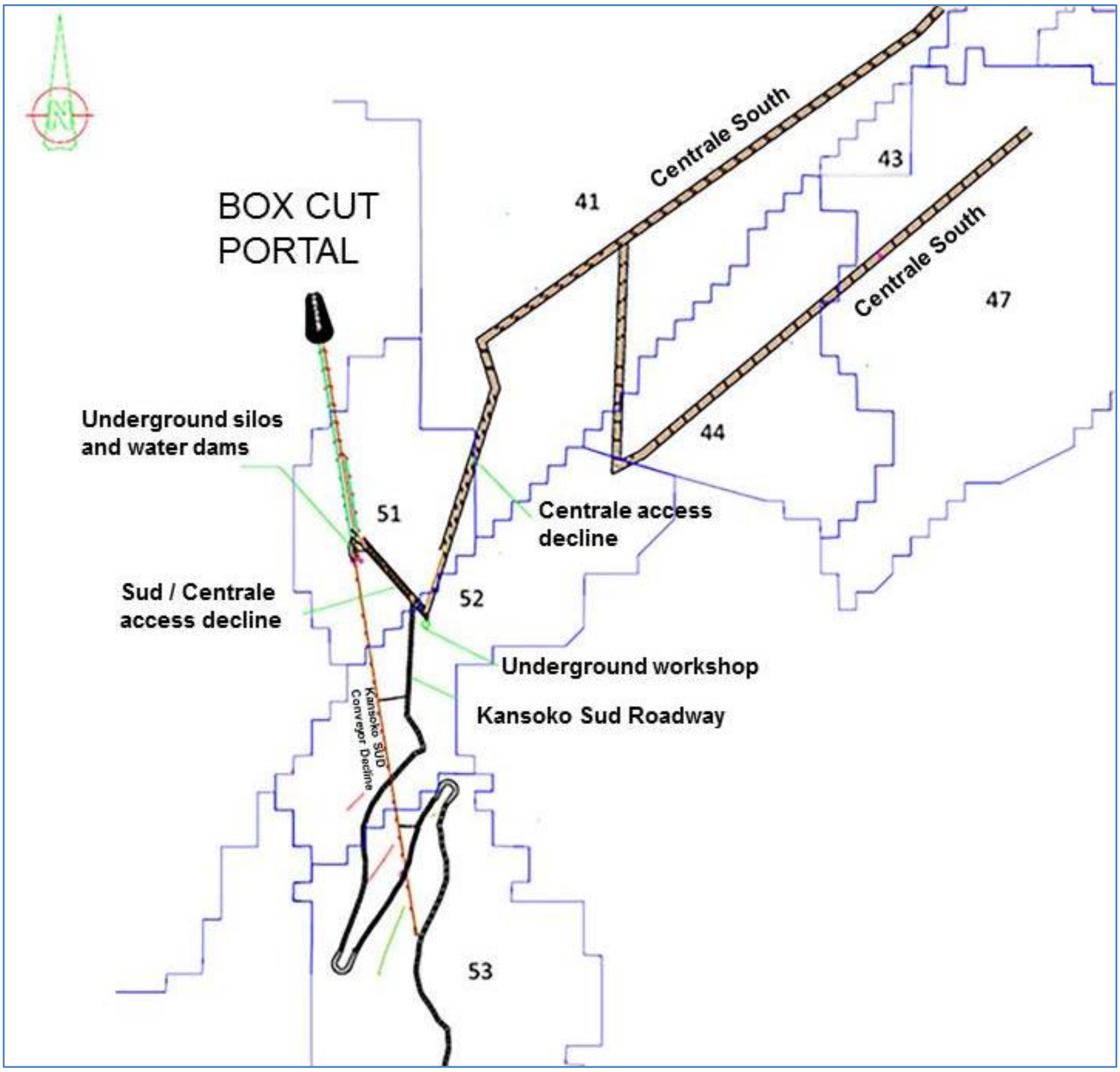


Figure by Ivanhoe, 2015.

The Kamoa ore body geometry indicates varying ore body thicknesses and slope. The ore body dips between 0° and 34° with an average dip of 16°. The thickness varies between 4.6 m and 5.6 m with an average vertical thickness of 5.1 m. The depth of the mining designed for the Mineral Reserve is between 170 m and 1,200 m below the surface.

A number of different mining methods were assessed for their suitability in mining the Kamoa deposit. The review indicated three methods could be used:

- Normal room and pillar mining for areas with a dip of less than 12°.
- Stepped room and pillar up to a dip of 34° and a depth of less than 500 m.
- Drift and Fill mining for depths in excess of 500 m.

The mining method currently planned for the Kamoa Project is a combination of stepped room and pillar (SR&P) and drift and fill (D&F).

Mine production is planned to be an average of 3 Mtpa ore over the current planned production period of 24 years.

The main areas of the ore body planned to be mined are the Kansoko Sud and Centrale areas. The Centrale area includes the Centrale North and Centrale South areas.

The Kamoa mine will be a mobile, trackless, mining operation. Bulk transport of the ore and waste will be carried out by a network of decline conveyor belt systems.

A trade-off study comparing trucking and conveying confirmed that conveyors would be less expensive than trucking for a large, long duration mining operation such as the Kamoa project.

The geometry of the Centrale ore body is more uniform than the Kansoko Sud ore body and therefore allows for the use of section conveyors. In the Centrale mining section, LHDs will load ore from the face onto the section conveyors which then transport ore onto the Centrale conveyor system. The Centrale conveying system consists of ten decline conveyors operating in a series arrangement transferring ore from the Centrale South and North mining areas onto the Kansoko Sud/Centrale decline access conveyor no. 1.

The geometry of the Kansoko Sud ore body, with varying dips in two planes, does not allow the use of section conveyors. In the Kansoko Sud mining section, LHDs will load ore from the mining face on to dump trucks. The ore is then trucked to ore passes which feed onto the Kansoko Sud decline conveyor system.

Both of these decline conveyor systems will convey ore into four underground storage silos. The underground storage silos have been designed to provide a surge capacity, equal to one day's production 8,500 t, to allow for periods when the main decline conveyor is inoperable.

The Main decline conveyor no.1, located at the bottom of the underground storage silos, will transport the total mine production of 3 Mtpa to the surface and onto the feed conveyor to the process plant.

There will be no underground rock crushing facilities. A trade-off study comparing underground crushers with hydraulic rock breakers found that hydraulic rock breakers were preferable, due to lower capital and operating costs and the ease of relocation in line with the advance of the mining face.

Figure 1.6 shows the position of the portal access in relation to the Kansoko Sud and Centrale mining areas. This portal location allows for earlier access to the shallower parts of the ore body 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.

Access to the underground mine is planned via a twin declines system from Kansoko Sud portal; located at the base of a 35 m deep boxcut. One decline will house the main conveyor and the other will be used as the service decline.

The service decline will allow the safe and reliable transport of mining personnel and equipment, provides alternate access to the conveyor decline and the reticulation of all services to and from the mine workings.

Additional major development required to access the mining areas includes the twin roadway system into the Kansoko Sud mining area and the triple decline system to the Centrale mining area.

The dimensions of these declines will be 5.5 m (W) x 6.0 m (H) based on the conveyor design, ventilation intake requirements and sizes of equipment.

Additional major underground infrastructure will include:

- Satellite underground workshop for daily and weekly servicing of the mobile production fleet.
- Diesel refuelling and lubrication storage tanks located in the satellite underground workshop.
- Intake ventilation raises (2 off x 5 m diameter), which combine with the main conveyor and service declines to supply the bulk of the ventilation to active areas in the Centrale and Kansoko Sud mining areas.

  Air cooling will be required from Year 7 onwards. Located on the surface the refrigeration and cooling plant will comprise a 7.2 MW bulk air cooler (BAC), a 7.0 MW refrigeration machine, an 8.8 MW cooling tower and ancillary equipment.

  The BAC will cool 300 m^3/s of intake air to be sent down the 5.0 m diameter ventilation raise feeding the Centrale South mining area.
- Exhaust ventilation raises (4 off x 3.5 m diameter, 2 off x 5 m diameter) will each be equipped with surface ventilation fans, to exhaust the designed total ventilation flow rate of 950 m^3/s.
- Explosives storage magazine, positioned central to the Kansoko Sud and Centrale mining areas.
- 11 kV electrical power reticulation and substations.
- Pumping systems and settling systems, designed to handle both mine service water and fissure water ingress.
- Permanent and mobile self-contained fire refuge stations.
- Fire detection and fire suppression systems on all conveyors.

For the mine production schedule SR&P stoping has been applied to the shallower areas and D&F to the deeper areas. Although the SR&P method is simpler and has lower operating costs since no fill material is required, as the mining progresses deeper, pillar sizes increase and the lower SR&P extraction ratios makes the cost of D&F cheaper than SR&P.

The SR&P mining method is suitable for orebodies dipping from 12° to 34° and with orebody thickness ranging between 3.5 m and 6.0 m. The orebody is developed in a series of horizontal steps. Haulage ramps are designed at an apparent dip of 8.5° which enables mechanised equipment to be used.

In the SR&P method the size of the remnant pillars increase with depth, to accommodate the increasing stresses, resulting in a reduction in the calculated extraction ratios.

The D&F mining method involves developing access drives on the strike of the orebody and spaced 75 m apart. Drifts will be developed on an apparent minor dip of 8.5° between the access drives.

Two drifts will be developed 5 m apart and holed into the adjacent drives. These drifts are then filled and the 5 m wide drift between the two filled drifts is mined. Paste fill is planned to be used and delivered from a surface facility.

While the D&F method enables higher calculated extraction ratios than the SR&P method, it will require the use of paste fill.

1.14.3 Alternative Mining Method

In parallel with the Kamoa 2016 PFS, an alternative mining method — controlled convergence room-and-pillar mining — was investigated for its suitability for use on the Kamoa deposit. Controlled convergence room-and-pillar mining does not require cemented backfill and instead pillars are stripped to allow the controlled convergence of the backs and floors. This potentially could provide significant cost savings as there is no requirement for cemented backfill and the Mineral Reserve extraction ratios are higher. The controlled convergence room-and-pillar mining method has been successfully implemented by KGHM at its copper mining operations in Poland for the past 20 years.

Ivanhoe engaged KGHM Cuprum R&D Centre Ltd. in early 2015 to study the applicability of this method to Kamoa. The results of the study were received toward the end of 2015 and indicate that the Kamoa deposit is suited to the application of the controlled convergence room-and-pillar mining method. The KGHM Cuprum study suggests that with the completion of the primary and secondary mining, the controlled convergence room-and-pillar mining method is capable of extracting 75% to more than 90% of the in-situ Mineral Reserve.

A sensitivity analysis to the base case Kamoa 2016 PFS economic analysis is shown in Table 1.12, where capital and operating costs associated with cemented backfill have been excluded and the in-situ Mineral Reserve extraction ratio has been increased.

Table 1.12 Mining Method Comparison - Overall Results

Item	Unit	SR&P and D&F (With Fill)	Controlled Convergence Room and Pillar (No Fill)
Ore Processed		PFS Base Case	PFS Sensitivity
Quantity Ore Treated	kt	71,893	71,893
Copper Feed Grade	%	3.86	3.86
Concentrate Produced			
Copper Concentrate Produced	kt (dry)	6,106	6,106
Copper Recovery	%	86.36	86.36
Copper Concentrate Grade	%	39.20	39.20
Contained Cu in Concentrate	Mlb	5,277	5,277
Key Financial Results			
Initial Capital	US$M	1,213	1,155
Mine Site Cash Cost	US$/lb Payable Cu	0.75	0.61
Total Cash Costs After Credits	US$/lb Payable Cu	1.48	1.35
Site Operating Costs	US$/t ore	53.22	43.54
After Tax NPV8	US$M	986	1,182
After Tax IRR	%	17.2	18.9
Project Payback Period	Years	4.6	4.3

Table 1.13 Mining Method Comparison Financial Results

After Taxation Net Present Value	(US$M)	SR&P and D&F (With Fill)	Controlled Convergence Room and Pillar (No Fill)
	Discount Rate	PFS Base Case	PFS Sensitivity
	Undiscounted	4,096	4,631
	4.0%	2,036	2,344
	6.0%	1,429	1,672
	8.0%	986	1,182
	10.0%	657	819
	12.0%	409	546
IRR	–	17.2%	18.9%
Project Payback (years)	–	4.6	4.3

1.15 Infrastructure

The project infrastructure includes power supply, tailings dams, communications, logistics, transport options, materials, water and waste water, buildings, accommodation, security, and medical services.

1.15.1 Electric Power

Electric power for the Kamoa Copper Project is planned to be sourced from three hydro power plants: the Koni, Mwadingusha, and Nzilo 1. A financing agreement with SNEL has been finalized for upgrading these plants to secure a long-term, clean, sustainable power supply to meet the requirements of Kamoa.

The Kamoa 2016 PFS initial capital cost of US$1.2 billion includes a US$104 million advance payment to SNEL to upgrade two of the hydro-power plants, Koni and Mwadingusha, to provide Kamoa with hydro-electric power during the initial phase of operations. The upgrading work is being led by Stucky Ltd. and the advance payment is expected to be recovered through a reduction in the power tariff once Kamoa is in operation. Kamoa initially will be powered by existing capacity on the national grid and on-site diesel generators, until upgrading work on the hydro-power plants has been completed.

1.15.2 Transportation

A phased logistics solution is proposed in the Kamoa 2016 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, Kamoa's production is expected to be transported by rail to the port of Lobito in Angola.

1.16 Recovery Methods

Ore with topsize 300 mm is conveyed from underground to surface stockpiles. A diverter is available to allow barren development 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.

Three variable speed apron feeders are available to recover ore from the stockpile and feed the 50 mm aperture scalping screen ahead of the primary crusher. Primary screen oversize is sent to the primary crusher feed bin and the undersize is sent direct to the secondary crushing feed bins. Primary crusher discharge joins secondary crusher discharge and is sent to the three sizing screen feed bins. Vibrating feeders control double deck screen feed. Oversize from both decks join the undersize from the primary screen in the secondary crusher feed bins. Vibrating feeders control cone crusher feed rate.

Sizing screen undersize (nominally -8mm) is sent to the mill feed stockpile. The mill feed stockpile is covered to minimise dust and has four vibrating feeders below it that feed ore onto the mill feed conveyor. The mill feed conveyor also has a grinding ball feeder discharging onto it to maximise primary milling power.

The primary ball mill is closed with a cyclone and produces a 150 µm P_{80} product. The primary cyclone overflow is sent to a linear screen to remove tramp oversize. The linear screen undersize gravitates to the secondary mill discharge sump.

The secondary ball mill circuit is also closed with a cyclone and generates a 53 µm P_{80} product that is sent to the flotation circuit. The flotation feed stream is sampled for accounting purposes.

After conditioning the flotation feed is pumped to the rougher flotation bank. Frother and collector are added at the feed box to the first rougher flotation cell and can be added to subsequent rougher and scavenger cells. Rougher concentrate from the first two cells is pumped to rougher cleaning cells. Rougher tails gravitates to the first scavenger cell.

Scavenger flotation takes place in a bank of nine cells. Scavenger concentrate forms part of regrind mill feed and scavenger tails forms part of the final tails stream.

Rougher cleaner concentrate is sent to rougher recleaner flotation and part 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 then to the regrind densifying cyclones. Densifying cyclone overflow reports directly to the regrind product tank and cyclone underflow is fed to the regrind mill. Mill discharge at 10 µm P_{80} reports to the Regrind Product Tank. The reground product is sampled and its particle size is continuously measured.

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 cleaner tails form part of final tails. 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 (coarse and fine) are sent to the thickener feed well. Thickener overflow reports to the concentrate thickener overflow tank, from where it is redistributed around the flotation circuit as spray water to maximise reagent reuse. Excess thickener overflow reports to the process water tank. Thickener underflow is pumped 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) mix in the tailings thickener feed tank then report to the thickener feed well. All tailings thickener overflow reports by gravity to the process water tank. Tailings thickener underflow is pumped to the tailings pumping tank and it is sampled for accounting purposes. Multistage slurry pumps send the slurry to the tailings storage facility.

1.17 Conclusions

1.17.1 Kamoa Project and Mineral Reserve

Given the significant Mineral Resource tonnage estimated and its large lateral extent, potential mining rates could increase, through operating in multiple mining areas and a series of production expansions to maximize extraction of the Mineral Resource and result in an increase in Mineral Reserves. A change in mining method to controlled convergence room and pillar, increase in production rate in both the mine and plant and the rail transport should all be considered in the next phase of study and project development.

1.17.2 Mineral Resource Estimate

Mineral Resources for the Project, have been estimated using core drill data, have been performed to industry best practices (CIM, 2003), and conform to the requirements of CIM Definition Standards (2014).

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

- Drill spacing is insufficient to determine if any local faulting exists, or the effects of any such faulting on lithology and grade continuity assumptions. Local faulting could disrupt the productivity of a highly-mechanised underground mining operation. In addition, the amount of contact dilution related to local undulations in the SMZ has yet to be determined. Ivanhoe plans to mitigate these risks with information derived from further infill drilling and an exploratory decline.

1.17.3 Underground Mining Methods

The controlled convergence room and pillar mining method described in the Alternative Mining Method discussion of the Kamoa 2016 PFS will allow for an increased extraction ratio and remove the need for fill for all mining areas.

An increase in mining rate appears reasonable given the multiple mining areas available on the mine footprint and a 4 Mtpa should be studied in the Feasibility Study. The ramp up periods for build up to full production could be increased by the need to open additional mining panels but this risk could be mitigated by the increased extraction ratio by adopting the controlled convergence room and pillar mining method.

Additional geotechnical data will be required to ensure the mining methods are suitable, the excavation and pillar dimensions are appropriate, and ground support requirements are adequate.

The Kamoa deposit appears to be relatively un-deformed (based on the current drill spacing and the lack of underground exposure), is continuously mineralised with good local continuity and flat to moderate dips. Additional infill drilling will be required to test for the presence of minor faults, which may result in changes in the roof and floor positions, increased dilution, and increased waste development. This drilling will also assist in validating local dip changes that may result in lower productivities. This drilling, along with further detailed mineralisation continuity studies, should help validate hanging and footwall continuity.

Additional hydrological data and studies will be required to confirm that groundwater inflows will not significantly impact underground costs and schedules. The water handling allowance included in the Kamoa 2016 PFS and the effects on ground conditions and productivities can then be further assessed. The thickness of the Kalahari sand in the overburden sequence may dictate the location of the ventilation raises and increase the costs of ventilation for the mine.

1.17.4 Recovery Methods

A two stage crush, two stage ball milling circuit prepares the ore to 53 µm P_{80} for flotation. Coarse copper mineralisation is recovered in a fast-floating circuit while middlings are reground to 10 µm to liberate the remaining copper ahead of scavenger flotation. Coarse and fine recleaner concentrates are filtered and bagged for export. Tailings are thickened and pumped to the TSF.

1.18 Recommendations

1.18.1 Further Assessment

With the additional data collection and the findings of the Kamoa 2016 PFS Ivanhoe is in a position to progress studies to a feasibility stage of assessment. Further study work should be undertaken to optimise the project production rate by considering concentrator capacities that are matched to the power supply availability, mine production and transport options. Based on the results of the Kamoa 2016 PFS, key changes that should be considered in the Feasibility Study are:

- Change the mining method to controlled convergence room and pillar stoping.
- Evaluate increased production up to 4 Mtpa from the proposed initial mining area using the controlled convergence room and pillar mining method and limited adjustments to the ore handling and ventilation systems.
- Continue to evaluate the options for rail transport to Lobito.

The following is a list of key activities for further work:

- Continue infill drilling programme to upgrade resource categorization, 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 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.
- 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.

1.18.2 Drill Programme

Amec Foster Wheeler has recommended a drill program that totals approximately 40,500 m, at a cost of about US$4.2 M. The program will have two objectives.

The first objective is exploration and infill drilling. Drillholes are proposed at Kakula for both hypogene and shallow supergene targets. Shallow drilling is recommended to be conducted at Kamoa Nord, Makalu NW, Kakula, Kansoko Nord, and around domes. In addition, deeper exploration drillholes are suggested to test other targets.

The second objective is to obtain additional information to support detailed studies. Additional engineering drillholes, including metallurgical drilling should be completed within the defined Kamoa Mineral Resource area.

1.18.3 Copper Mineralogy Characterisation

Proportions of the major copper minerals, chalcopyrite, bornite and chalcocite, are variable throughout the orebody. In addition, in the supergene ores, the amount of copper in non-floating minerals is also highly variable. Product grade is determined by the mineral mix in the concentrate while recovery is mainly determined by the proportion of copper in non-floating minerals. Better methods of determining the recoverable mineral mix and the amount of non-floating copper are needed if prediction of concentrate grade and recoveries is to be successful.

2 INTRODUCTION

2.1 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 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 and Platreef Projects from discovery to production is a key near-term objective. Ivanhoe Mines owns a 49.5% share interest in Kamoa Holding Limited (Kamoa Holding), an Ivanhoe subsidiary that presently owns 95% of the Kamoa Copper Project. Zijin Mining Group Co., Ltd. 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 was transferred to the DRC government on 11 September 2012, for no consideration, pursuant to the DRC Mining Code. Ivanhoe also has offered to transfer an additional 15% interest to the DRC government on terms to be negotiated. Constructive and cordial negotiations between Ivanhoe Mines, Zijin and senior DRC government officials have been continuing in this regard.

2.2 Terms of Reference

The Kamoa 2016 PFS is an Independent Technical Report on the Kamoa Copper Project (the Project) prepared for Ivanhoe Mines Limited (Ivanhoe) as part of the Ivanhoe strategy for development of the Project.

The Kamoa 2016 PFS is a Prefeasibility Study with an effective date of 29 March 2016 that has been prepared using the June 2011 edition of Canadian National Instrument 43-101 Standards of Disclosure for Mineral Projects.

The following companies have undertaken work in preparation of the Kamoa 2016 PFS:

- OreWin: Overall report preparation, underground mining, open pit potential and financial model.
- Amec Foster Wheeler: Geology, Drillhole data validation, and Mineral Resource estimation.
- SRK Consulting Inc.: Mine geotechnical recommendations.

- Kamoa Copper SA and Worley Parsons.: Underground mine planning and design.
- MDM and Amec Foster Wheeler: Process and infrastructure.
- Golder Associates: paste backfill, hydrology, hydrogeology, and geochemistry.
- Epoch Resources (Pty) Ltd: Tailings Storage Facility (TSF).

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, 1.2, 1.3, 1.4, 1.13, 1.14, 1.17.1, 1.17.3, 1.18.1; Section 2; Section 3; Section 4, Section 5; Section 10.7, Section 15; Sections 16.2, 16.3, 16.4, 16.5, 16.6; Section 19; Section 20; Sections 21.1, 21.2, 21.8; Section 22; Section 23; Section 24; Sections 25.2, 25.3, 26.1, 26.4; Section 27.
- Dr Harry Parker, SME Registered Member (2460450), Technical Director, Amec Foster Wheeler was responsible for: Sections 1.5 to 1.9, 1.11, 1.12, 1.17.2, 1.18.2; Section 2; Section 3; Section 6; Section 7; Section 8; Section 9; Sections 10.1 to 10.6, 10.9 to 10.11; Sections 11.1, 11.2, 11.4 to 11.11; Section 12, Section 14; Section 25.1, Sections 26.2, 26.3; Section 27.
- Gordon Seibel, SME Registered Member (2894840), Principal Geologist, Amec Foster Wheeler was responsible for: Sections 1.5 to 1.9, 1.11, 1.12, 1.17.2, 1.18.2; Section 2; Section 3; Section 6; Section 7; Section 8; Section 9; Sections 10.1 to 10.6, 10.9 to 10.11; Sections 11.1, 11.2, 11.4 to 11.11; Section 12, Section 14; Section 25.1, Sections 26.2, 26.3; Section 27.
- William Joughin, FSAIMM (55634), employed by SRK Consulting (South Africa) (Pty) Ltd as Principal Consultant, was responsible for: Section 1.14.1; Section 2; Section 16.1.
- Dean David, B AppSc (Metallurgy), FAusIMM. (102351), Technical Director, Amec Foster Wheeler was responsible for: Sections 1:10, 1:15, 1:16, 1.17.4, 1.18.3; Section 2; Section 3, Section 10.8; Section 11.3; Section 13; Section 17; Section 18; Sections 21.3 to 21.7, 21.9, 21.10; Section 25.1; Section 26.5; Section 27.

2.4 Site Visits and Scope of Personal Inspection

Site visits were performed as follows.

Mr Bernard Peters visited the site from 15 February 2010 to 17 February 2010 and again from 27 April 2010 to 30 April 2010; on 15 November 2012 and from 12 September 2015 to 14 September 2015. The site visits included briefings from Ivanhoe geology and exploration personnel, site inspections of 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 Project site.

Dr Harry Parker visited the Kamoa Project from 1 to 3 May 2009, from 27 to 30 April 2010, and again from 12–14 November 2012. 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 areas, and discussions of geology and mineralisation interpretations with Ivanhoe's staff.

Mr Gordon Seibel visited the Project from 9 to 10 February 2011, from 5 to 8 November 2011, from 12–14 November 2012 and again from 18–22 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 the Discovery Target.

Mr William Joughin has not visited the site but the site has been visited by personel 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.

Table 2.1 SRK Site Visits

Person	Dates	Purpose
Jarek Jakubec	April 27 to May 1 2010	Initial project geotechnical review
Wayne Barnett	21 to 25 July 2010	Review progress in geotechnical characterization 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 characterization 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 Pre-feasibility geotechnical study.
Desmond Mossop	18 t0 20 November 2014	Geotechical Review of the Kamoa Box-cut, Portals and Decline Ground Control
Rory Bush	21 to 26 June 2015	Geotechical Review of the Kamoa Decline Ground Support Recommendations

Mr Dean David visited the Kamoa Project from 27 to 30 April 2010, and again from 13 to 15 April 2011. During both visits he conducted inspections 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 imminent metallurgical testwork programs.

2.5 Effective Dates

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

- Date of the last supply of information on the ongoing drilling programs and the latest database copy: 5 November 2015. Information available from this data supply was used to validate the geological model.
- Date of drillhole database close-out date for updated Mineral Resource estimate: 5 May 2014.
- Date of Mineral Resource update for mineralisation amenable to underground mining methods: 5 May 2014.

- Date of the supply of legal information supporting mineral tenure that supports the Kamoa 2016 PFS: 25 January 2016.

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 2016 PFS, 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 legal experts for this information through the following document:

- 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 Ltd., 2016: Kamoa Copper Project: unpublished letter prepared by representatives of Ivanhoe for OreWin, 29 March 2016.

This information was used in Section 4.3 of the Report and Section 14.11 for assessment of reasonable prospects of economic extraction.

The QPs have also fully relied upon, and disclaim responsibility for, information supplied by Ivanhoe staff from legal experts for information relating to mineral tenure, ownership of the Project area, underlying property agreements and permits through the following documents:

- Geraghty, L., 2010: PR 702 to 706: Copies of Original Signed Permit Certificates: unpublished internal email to Amec Foster Wheeler, dated 8 September 2010.
- Ivanhoe Mines Ltd., 2016: Kamoa Copper Project: unpublished letter prepared by representatives of Ivanhoe for OreWin, 29 March 2016.

This information was used in Section 4 of the Report, and Section 14.11 for assessment of reasonable prospects of economic extraction.

3.2 Surface Rights

The QPs have fully relied upon, and disclaim responsibility for, information supplied by Ivanhoe's Kinshasa-based lawyer for information relating to payment of land and surface rights taxes and payment due dates for 2009–2012 through the following documents:

- 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 Ltd., 2016: Kamoa Copper Project: unpublished letter prepared by representatives of Ivanhoe for OreWin, 29 March 2016.

This information was used in Section 4 of the Report, and Section 14.12 for assessment of reasonable prospects of eventual economic extraction.

The QPs have fully relied upon, information supplied by Ivanhoe staff and experts retained by Ivanhoe for information relating to the status of the current Surface Rights as follows:

- Rogers, T., 2010: Land tax and Surface Rights Fees: unpublished internal email to Amec Foster Wheeler, dated 8 September 2010.
- African Mining Consultants, 2009: Greater Kamoa Project, The Democratic Republic of Congo, Environmental Impact Assessment Scoping Study: unpublished report prepared by African Mining Consultants for African Minerals (Barbados) Ltd. Sprl, June 2009.
- Broughton, D., 28 May 2012: Email confirming tenure, permits and payments.
- Ivanhoe Mines Ltd., 2016: Kamoa Copper Project: unpublished letter prepared by representatives of Ivanhoe for OreWin, 29 March 2016.
- Geraghty, L., 2013: Surface Rights Payment Kamoa: unpublished internal emails to Amec Foster Wheeler, 30 September 2013 with attachments.

This information was used in Section 4 of the Report, and Section 14.11 for assessment of reasonable prospects of 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 Amec Foster Wheeler QPs have fully relied upon, and disclaim responsibility for, information derived from such experts through the following document:

- Ivanhoe Mines Ltd., 2016: Kamoa Copper Project: unpublished letter prepared by representatives of Ivanhoe for OreWin, 29 March 2016.

This information was used in Section 20 of the Report.

- African Mining Consultants, 2009d: Greater Kamoa Project, The Democratic Republic of Congo, Environmental Impact Assessment Scoping Study: unpublished report prepared by African Mining Consultants for African Minerals (Barbados) Ltd., Sprl, dated June 2009.

This information was used in Section 20 of the Report and Section 14.11 for assessment of reasonable prospects of economic extraction.

- Ivanhoe Mines Ltd., 2016: Kamoa Copper Project: unpublished letter prepared by representatives of Ivanhoe for OreWin, 29 March 2016.
- 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.

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 Congo, dated 01 March 2016.
- Ivanhoe Mines Ltd., 2016: Kamoa Copper Project: unpublished letter prepared by representatives of Ivanhoe for OreWin, 29 March 2016.

This information was used in Section 20 of the Report and Section 14.11 for assessment of reasonable prospects of economic extraction.

4 PROPERTY DESCRIPTION AND LOCATIONS

The Kamoa Project is situated in the Kolwezi District of Lualaba Province, DRC. The Kamoa Project is located approximately 25 km west of the town of Kolwezi, and about 270 km west of the provincial capital 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

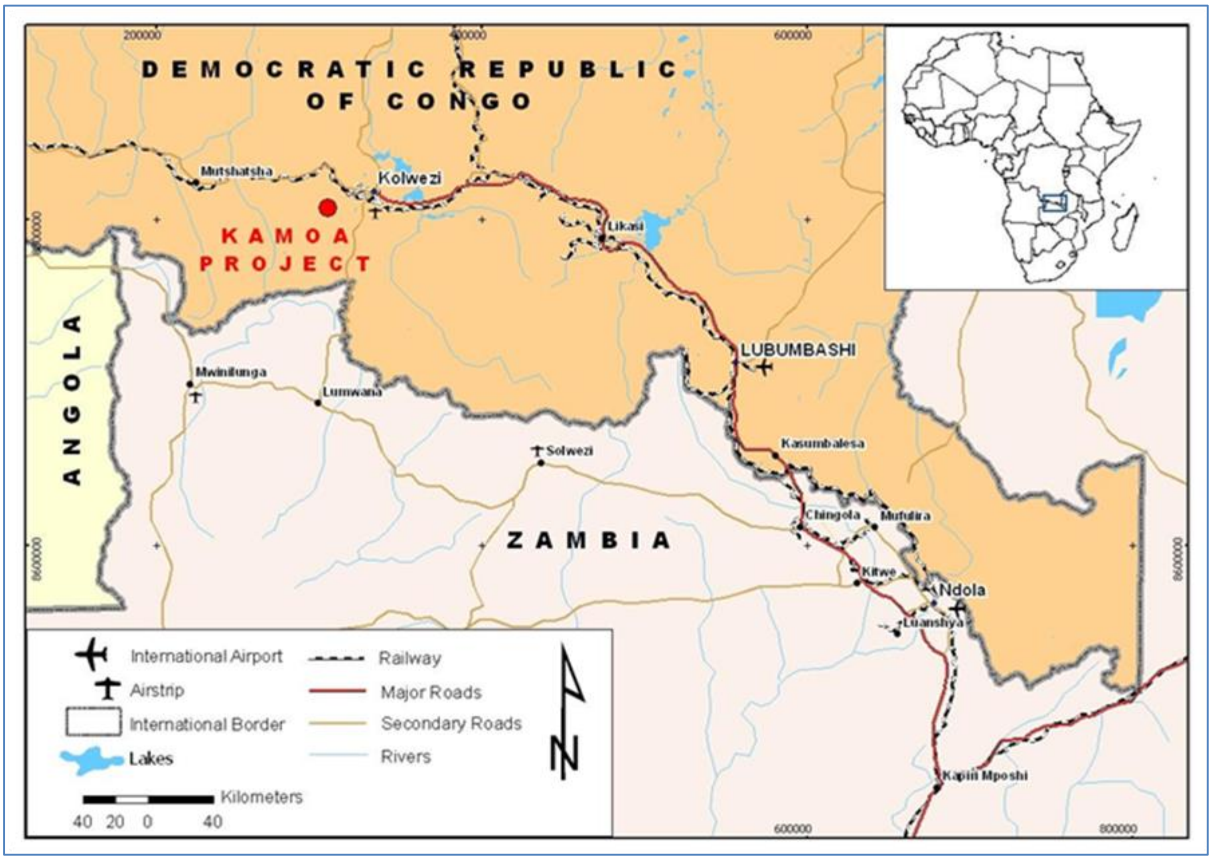


Figure by Ivanhoe, 2011

4.1 Project Ownership

Ivanhoe Mines owns a 49.5% share interest in Kamoa Holding Limited (Kamoa Holding), an Ivanhoe subsidiary that presently owns 95% of the Kamoa Copper Project. Zijin Mining Group Co., Ltd. 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 was transferred to the DRC government on 11 September 2012, for no consideration, pursuant to the DRC Mining Code. Ivanhoe also has offered to transfer an additional 15% interest to the DRC government on terms to be negotiated. Ivanhoe has reported that constructive and cordial negotiations between Ivanhoe Mines, Zijin and senior DRC government officials have been continuing in this regard.

4.2 Property and Title in the Democratic Republic of Congo

4.2.1 Introduction

A summary of the mining history of the Katangan region is presented below, and is adapted from André-Dumont (2013), and 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 nationalized 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 Project.

4.2.2 Mineral Property Title

The following summary on mineral title is adapted from André-Dumont (2013), and the 2002 DRC Mining Code.

All mineral rights in the DRC are state-owned, and the holder of mining rights gains ownership of the mineral products for sale.

The main legislation governing mining activities is the Mining Code (Law No. 007/2002) dated 11 July 2002 (the 2002 Mining Code). The applications of the Mining Code are provided by the Mining Regulations enacted by Decree No. 038/2003 of 26 March 2003 (the 2003 Mining Regulations). The legislation incorporates environmental requirements.

The Ministry of Mines regulates the Mining Registry, Directorate of Mines, and the Geological Directorate 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:

- President of the DRC: can enact mining regulations, exercises powers by decree.
- Minister of Mines: jurisdiction over the granting, refusal and cancellation of mining rights, can exercise powers by decree.
- Mining Registry: (Cadastre Minier; a public entity supervised by the Minister of Mines and the Minister of Finance) conducts administrative proceedings concerning the application for, and registration of, mining rights, as well as the withdrawal, cancellation and expiry of those rights.
- Directorate of Mines: responsible for inspecting and supervising mining activities with regard to safety, health, work practices, production, transport, sales, and social matters.
- Mining Environment Protection Department: responsible for definition and implementation of mining regulations concerning environmental protection, and technical evaluation of rehabilitation and mitigation plans; supervises and reviews environmental impact studies, and environmental management plans.

Under the 2002 Mining Code, mining rights are regulated by Exploration, Exploitation, Small-Scale Exploitation, and Tailing 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 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) of its share capital 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 consisting of entire contiguous quadrangles subject to the limits relating to the borders of the National Territory and those relating to reserved prohibited areas and protected areas as set forth in the 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. Perimeters are indicated on 1:200,000 scale maps that are maintained by the Mining Registry.

Within two months of issuance of a mining or quarry Exploitation Permit, the holder is expected to survey the perimeter. The survey consists of placing a survey marker at each corner of the perimeter, and placing a 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 2013 Decrees

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 include more onerous formalities and conditions for obtaining mining rights. The changes to the 2002 Mining Code have not been finalized and are yet to be enacted. In February 2016 the DRC Minister of Mines announce that the the current code will be retained.

During 2013, the DRC Minister of Mines passed two new decrees. The first, dated 5 April 2013, bans the export of copper and cobalt concentrates, and includes a reduced moisture content requirement for concentrates intended for export. Moratoriums were granted on the decrees and the next moriatorium expires to 31 December 2016. The decree also limits the costs that are deductible against mining royalty assessments.

The second decree, dated 17 April 2013, requires mining operators to only use Congolese companies for supplies and works, though it has an exemption for "external expertise or a qualified foreign company" (McGuireWoods, 2013). There has been objection to this decree that argues it is contrary to the 2002 Mining Code. It is expected that the exemption for external expertise would be applied to Kamoa because of the nature of the project.

4.2.4 Exploration Permits

Exploration Permits as defined in the 2002 Mining Code allow the holder to:

> "carry out mineral exploration work for mineral substances classified as mines, substances for which the licence has been granted, and associated substances if the holder applies for the licence to be extended to include 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. No company can hold more than 50 Exploration Permits in the DRC, and the area of 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 surrender 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 surrender requirements.

Additional criteria can apply to mineral Exploration Permits under the 2002 Mining Code:

> "The holder is authorized to take samples of the mineral substances within the Perimeter indicated on his Mineral Exploration Licence in order to carry out analyses or industrial assays in the laboratory or plant of his choice".

However, under the 2002 Mining Code:

> "The holder of a Mineral Exploration Licence is required to submit to the Geology Directorate of the Ministry of Mines a duplicate sample of all of the samples or samples batches taken within the Perimeter of his 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 a permit is dependent on a number of factors that are defined in the 2002 Mining Code, including:

a) Demonstration of the existence of a deposit which can be economically exploited, by presenting a feasibility study under the laws of the DRC, accompanied by a technical framework plan for the development, construction, and exploitation work for the mine.

b) Demonstration of the existence of the financial resources required for the carrying out of his 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 is closed. This plan specifies each type of financing, the sources of planned financing and justification of their possible availability.

c) Obtain in advance the approval of the project's EIS[1] and the EMPP[2] .

d) Transfer to the Government 5% of the shares in the registered capital of the company applying for the licence. These shares are free of all charges and cannot be diluted.

The Exploitation Permit, as defined in the 2002 Mining Code, allows the holder the exclusive right to:

> "carry out, within the Perimeter over which it has been granted, and during its term of validity, exploration, development, construction and exploitation works in connection with the mineral substances for which the licence has been granted, and associated substances if he has applied for an extension."

In addition, it entitles, without restriction, the holder to:

a) Enter the Exploitation Perimeter to conduct mining operations.

b) Build the installations and infrastructures required for mining exploitation.

c) Use the water and wood within the mining Perimeter for the requirements of the mining exploitation, complying with the requirements set forth in the EIS and the EMPP.

d) Use, transport and freely sell his products originating from within the Exploitation Perimeter.

e) 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.

f) Proceed to carry out works to extend 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, show that the holder has:

- Not breached the holder's obligations to maintain the validity of the licence.
- Demonstrated the fact that the deposit is not exhausted by updating the feasibility study under the laws of the DRC.
- Demonstrated the existence of the financial resources required to continue to carry out his project, according to the financing plan and exploitation work in the mine, as well as the rehabilitation plan for the site when the mine will be closed. This plan details each type of financing planned and reasons of its possible availability.
- Obtained the approval for the updating of the EIS and EMPP.

[1] EIS = Environmental Impact Statement
[2] EMPP = Environmental Management Protection Plan

- 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 Mining Regulations, a mining rights holder must pay a levy on the total surface area of a mining title (Article 238 of the 2002 Mining Code). Levies are defined in the 2003 Mining Regulations on a per hectare basis, and increase on a sliding scale for each year that the mining title is held, until the fourth 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 Mining Registry and the Ministry of Mines, and payable annually to the Mining Registry, 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 DRC State has exclusive rights to the land land, (Land Law No. 73-021 dated 20 July 1973), but 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 states that subject to any rights of third parties over the surface concerned, the holder of an exploitation mining right has, with the authorisation of the governor of the province concerned, and on the advice of the Administration of Mines, the right to occupy within a granted mining perimeter the land necessary for mining and associated industrial activities, including the construction of industrial plants and dwellings, water use, dig canals and channels, and establish means of communication and transport of any type.

Any occupation of land that deprives surface right holders from using the surface, or any modification rendering the land unfit for cultivation, entails an obligation on the part of the mining rights holder to pay fair compensation to the surface right holders. The mining rights holder is also liable for damage caused to the occupants of the land in connection with any mining activity, even if such an activity has been properly permitted and 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 or an exploration permit, a company must complete an environmental impact study to be filed with the environmental plan.

On approval, the applicant must provide security for rehabilitation by means of a bank guarantee. Funds posted as security are not at the disposal of the DRC government and are to be used for the rehabilitation of a mining site.

Exploration Permit

Each Exploration Permit in the DRC requires an exploration impact and remediation plan (PAR). 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. Environmental reviews are required to be carried out at regular intervals, in support of annual environmental reporting requirements.

Exploitation Permit

Environmental obligations for conversion of an Exploration Permit to an Exploitation Permit under the Mining Code require the preparation of an environmental impact study and an environmental management plan for a development project, both of which must be updated if a renewal of an Exploitation Permit is sought. The Mining Code has provision for a biennial environmental audit. If a company does not pass this audit, it may lose its permit.

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 environmental impact study and environmental management plan.

4.2.8 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.

4.3 Mineral Tenure

The Kamoa 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 license (exploration permit 703 covers an area of 12.74 km^2).

Prior to grant, the Exploitation Permits were appropriately surveyed in accordance with DRC requirements. 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 (km²)
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	173.2
13026	20 Aug 2012	19 August 2042	Silver, Bismuth, Cadmium, Cobalt, Copper, Iron, Germanium, Nickel, Gold, Palladium, Platinum, Lead, Rhenium, Sulphur and Zinc.	202	171.5
Sub Total					**397.4**
Exploration Permit (PR) No.	**Grant Date**	**Expiry Date**	**Mineral/Metal Rights Granted**	**Number Cadastral Squares (Quadrangles)**	**Area (km²)**
703	11 Nov 2003	10 May 2020	Base, Precious, Platinum Group Metals, Pegmatite Minerals, Diamonds and Gemstones	15	12.7
Sub Total					1.7
Total					**410.1**

Figure 4.2 Project Tenure Plan

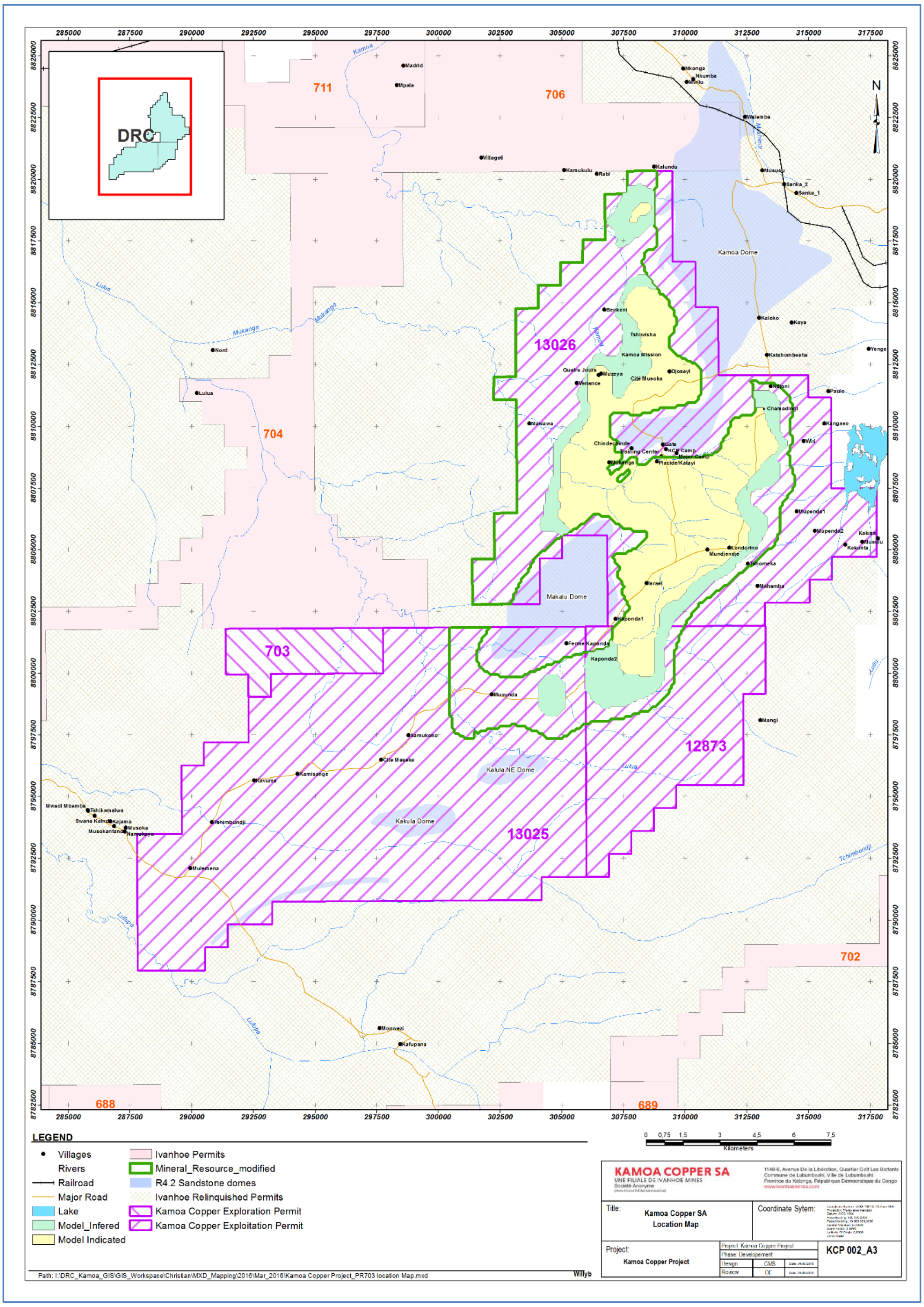


Figure by Ivanhoe, 2016.

Ivanhoe advised the QPs that the company 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 2015 were made for the three Exploitation Permits and Exploration Permit 703, and that the company has paid the required fees for 2015.

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, Ivanhoe holds no surface rights in the Project area. No title deed holders 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.

4.5 Royalties and Encumbrances

A discussion of the royalty considerations payable to the DRC Government is included in Section 4.2.8.

4.6 Property Agreements

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

4.7 Permits

Project permitting considerations are discussed in Section 20.

4.8 Environmental Liabilities

Environmental risks and liabilities to the Project are discussed in Section 20.

4.9 Social License

Considerations of social licence in support of Project development activities are discussed in Section 20.

4.10 Comments on Section 4

The information discussed in this section supports the assumptions and results in the Technical Report:

- Information from legal experts supports that the mining tenure held is valid.
- Surface rights have yet to be obtained. The Project area is sparsely inhabited, and Ivanhoe's investigations to date not identified any title deed holders in the area identified for mining infrastructure. Surface rights are currently being negotiated with

communities in compliance with DRC law.

- Ivanhoe will need to apply for additional specialist permits as appropriate to the jurisdiction to allow mining operations.
- Ivanhoe has offered to sell the DRC Government an additional 15% interest in the Project on commercial terms to be negotiated.
- The proposed amendments to the 2002 Mining Code contained in the draft law include more onerous formalities and conditions for obtaining mining rights. The changes to the 2002 Mining Code have not been finalized and are yet to be enacted.
- To the extent known, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the property.

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 deposit, 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. The route is a combination of tarred and gravel roads, which have recently been refurbished and are 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.

5.1.3 Rail

Until 2012, the rail line of approximately 740 km between Ndola (border with DRC) and 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 forecasted 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.

The condition of and access to the current rail infrastructure in the DRC make rail a less viable option for inbound project logistics. A more detailed discussion of the rail networks in Section 18.

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. At Kamoa the average annual temperatures vary between 16°C and 28°C, with the average being 20.6°C. Winds at the Kamoa Copper Project (KCP) 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 Resources and Infrastructure

The Project is currently isolated from public infrastructure. Infrastructure on-site is currently limited to support for the exploration activities. Ivanhoe has established a regional exploration base camp in Kolwezi because of the availability of power and communications in the town.

Exploitation of the Kamoa deposit 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. The infrastructure requirements envisaged in the Kamoa 2016 PFS are discussed in Section 18.

5.4 Power

Power for the Kamoa 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 Kamoa 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 Democratic Republic of Congo'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). Since this study the production rates has been amended to 3 Mtpa. Further studies will be required to assess these more modest power requirements in the initial phase of the project. 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 megawatts (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 average metres above sea level (amsl), with current exploration activities in areas of elevation from 1,450 m to 1,540 m amsl. The local topography of the Project is affected by the drainage catchments of the Mukanga, Kamoa, and Lulua Rivers and the Kalundu, Kansoko, and Kabulo Streams.

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

The Project is generally well vegetated with Central Zambezian Miombo woodland, 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 Foster Wheeler for the Project in 2009, and the estimate has been updated in 2010, 2011, 2012, and 2014.
Preliminary Economic Assessments were prepared in 2012 and in 2013.

7 GEOLOGICAL SETTING AND MINERALISATION

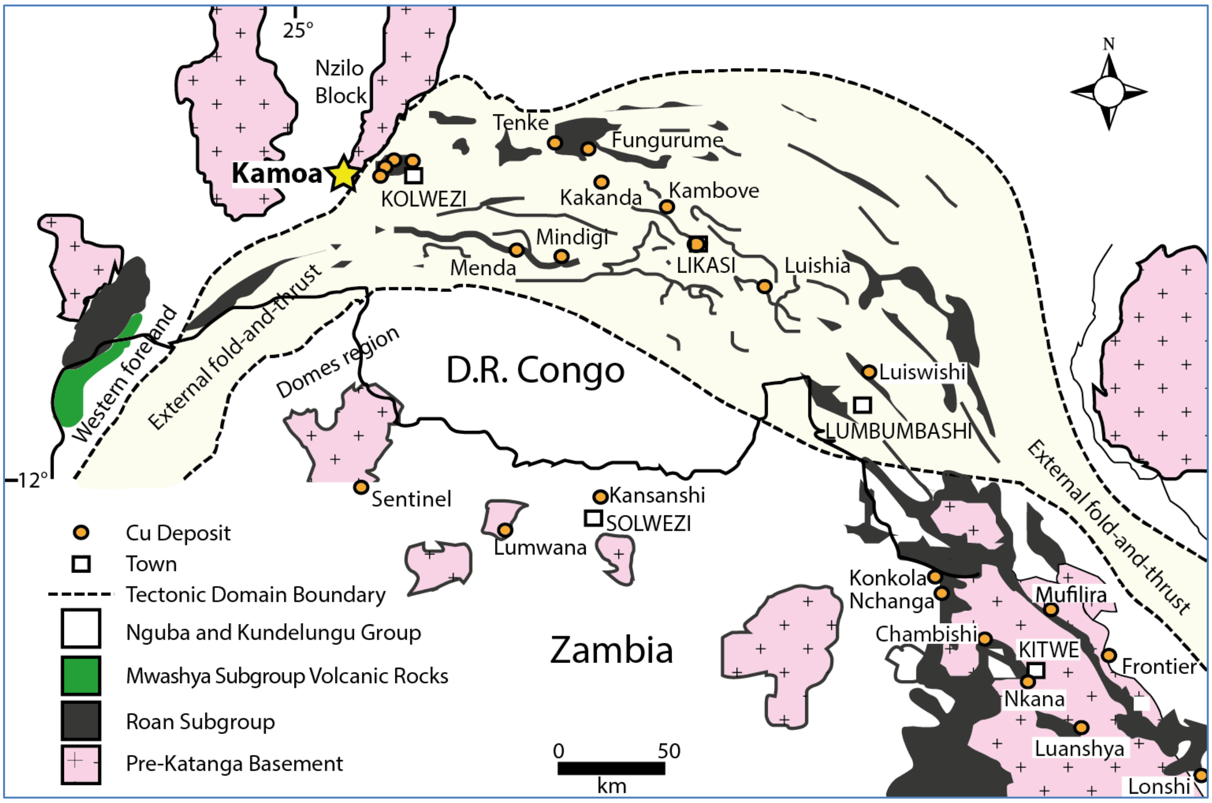
The discussion in this section has been prepared from published papers on regional geology as cited, particularly Schmandt et al (2013), and is also based on discussions with, and presentations made by, Ivanhoe personnel (David Broughton, David Edwards, and George Gilchrist) and African Mining Consultants (Thomas Rogers and Steffen Kalbskopf). The presentation made in the previous Technical Report (Peters et al., 2013) has been updated for work done by Ivanhoe staff during the last two years.

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 which 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 (Kamoa 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 to 1,300 Ma) is known.

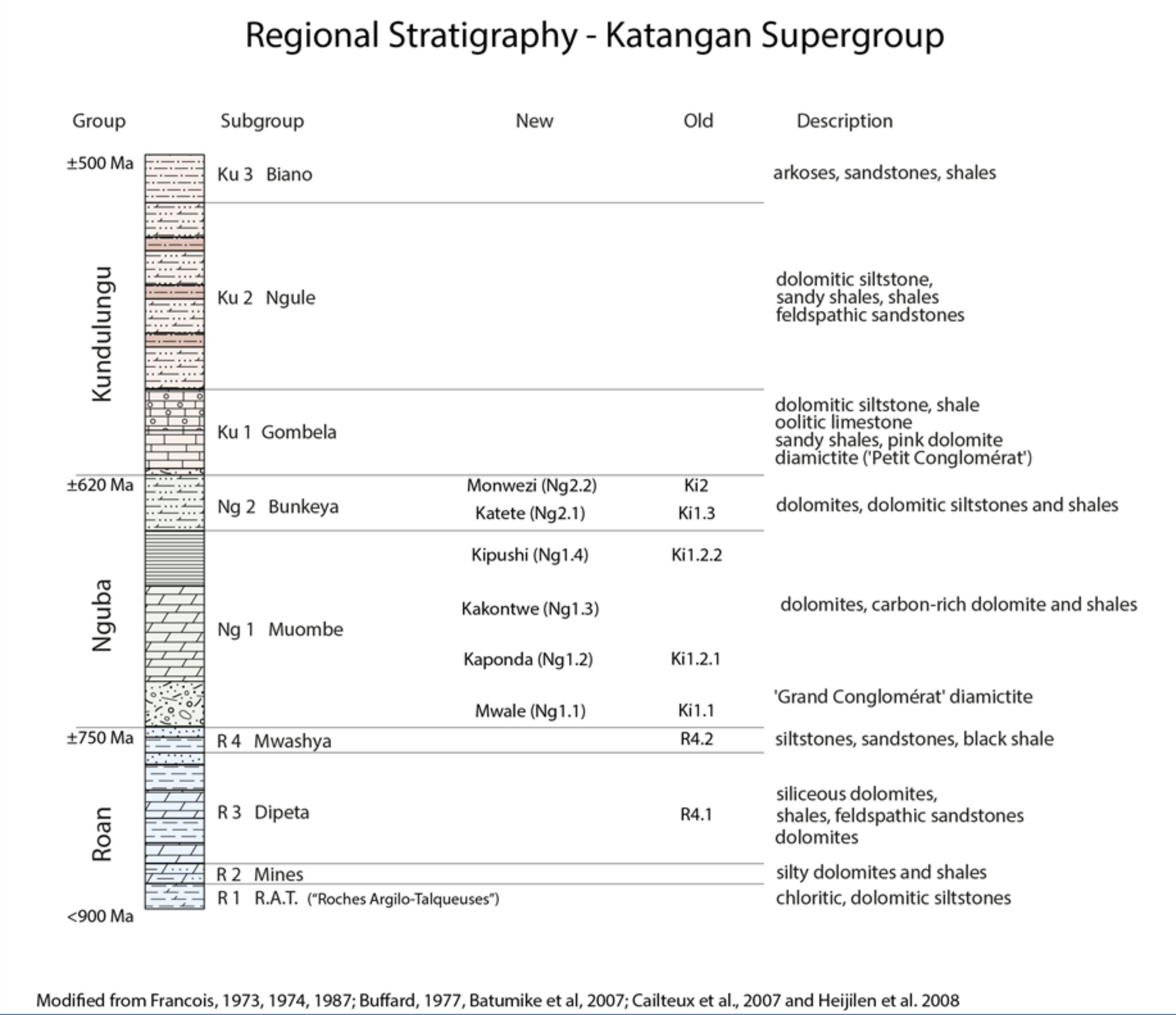
Figure 7.1 Geological Setting Central African Copperbelt



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



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 recognized 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 characterized 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 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 megaframents 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 Project area is considered by Ivanplats to form part of the Foreland.

Figure 7.3 Location of the Kamoa Copper Project in Relation to the Regional Geology of the Kamoa and Kolwesi area

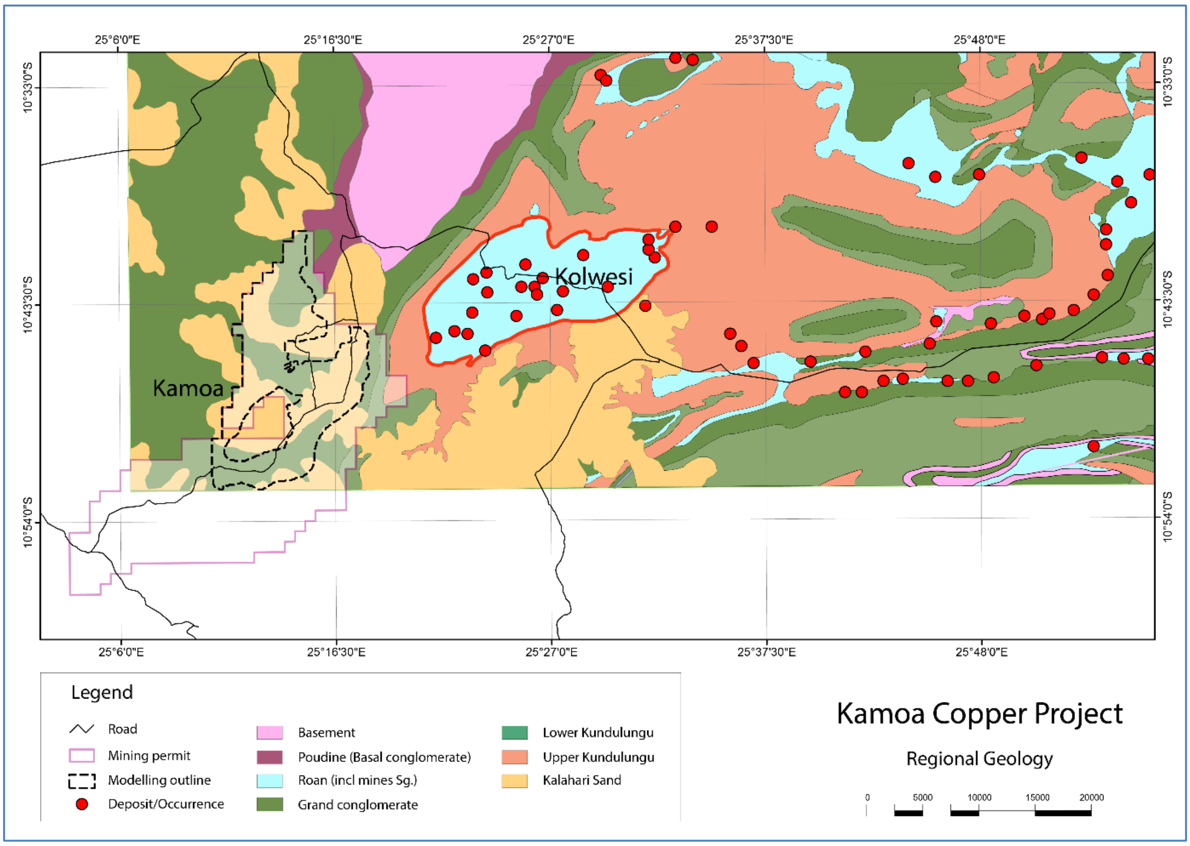


Figure provided by Ivanhoe, 2016.

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 Project is located in a broadly-folded terrane centered on the Kamoa and Makalu domes between the West Scarp Fault and Kansoko Trend. The domes form erosional windows exposing the redox boundary between the underlying hematitic (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 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).

Figure 7.4 Prospect Areas within the Combined Exploitation Permits

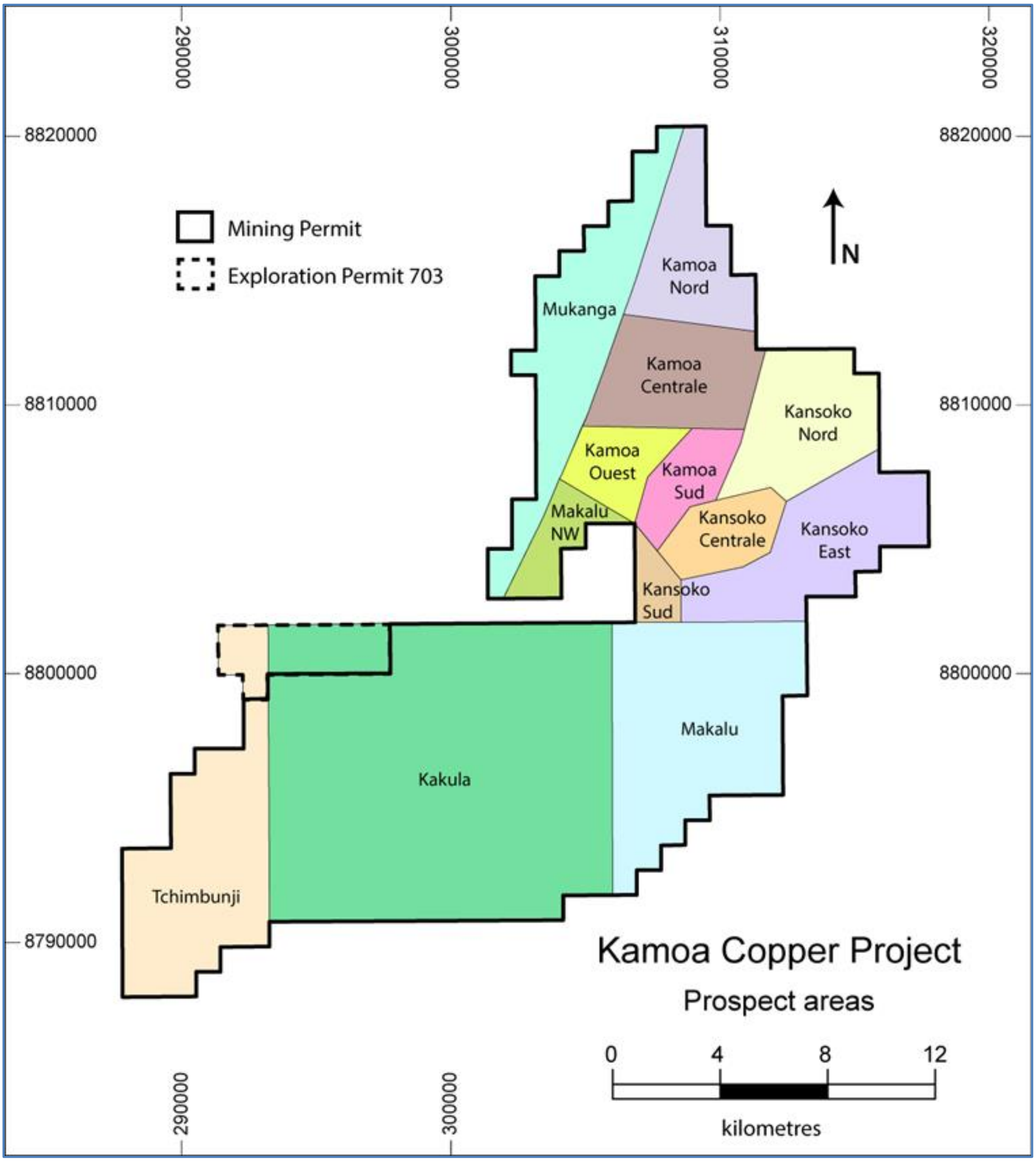


Figure provided by Ivanhoe, 2016

7.3 Deposit Description

7.3.1 Stratigraphic Sequence

Within the Project area, a localized stratigraphy has been recognized, defining greater detail for the Lower Kundulungu Group, refer to Figure 7.5. 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 hematite 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 7.6. 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 7.7, 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 7.7. Dropstones (also shown in Figure 7.7) 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).

Figure 7.5 Local Stratigraphy for the Kamoa Copper Project

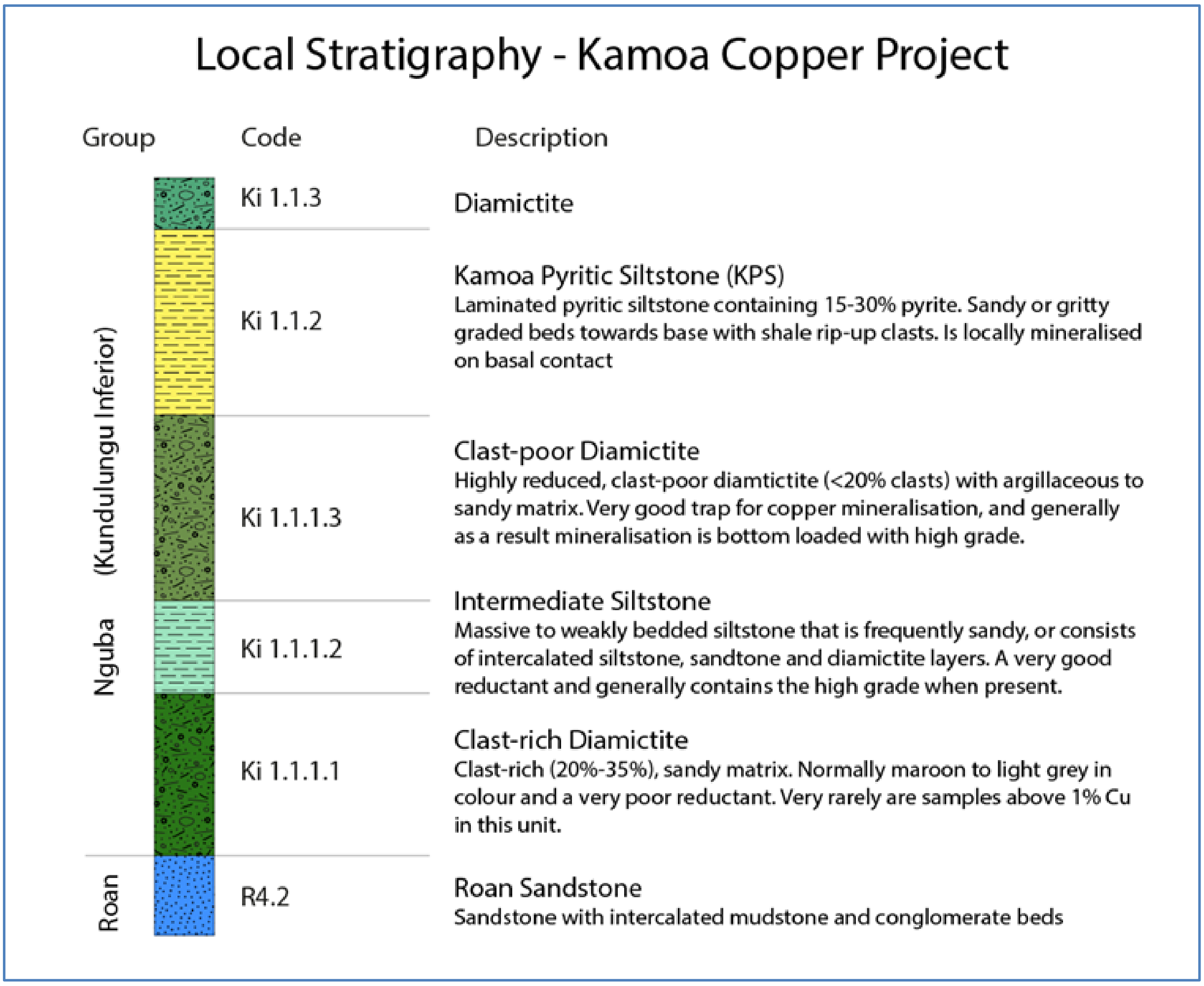
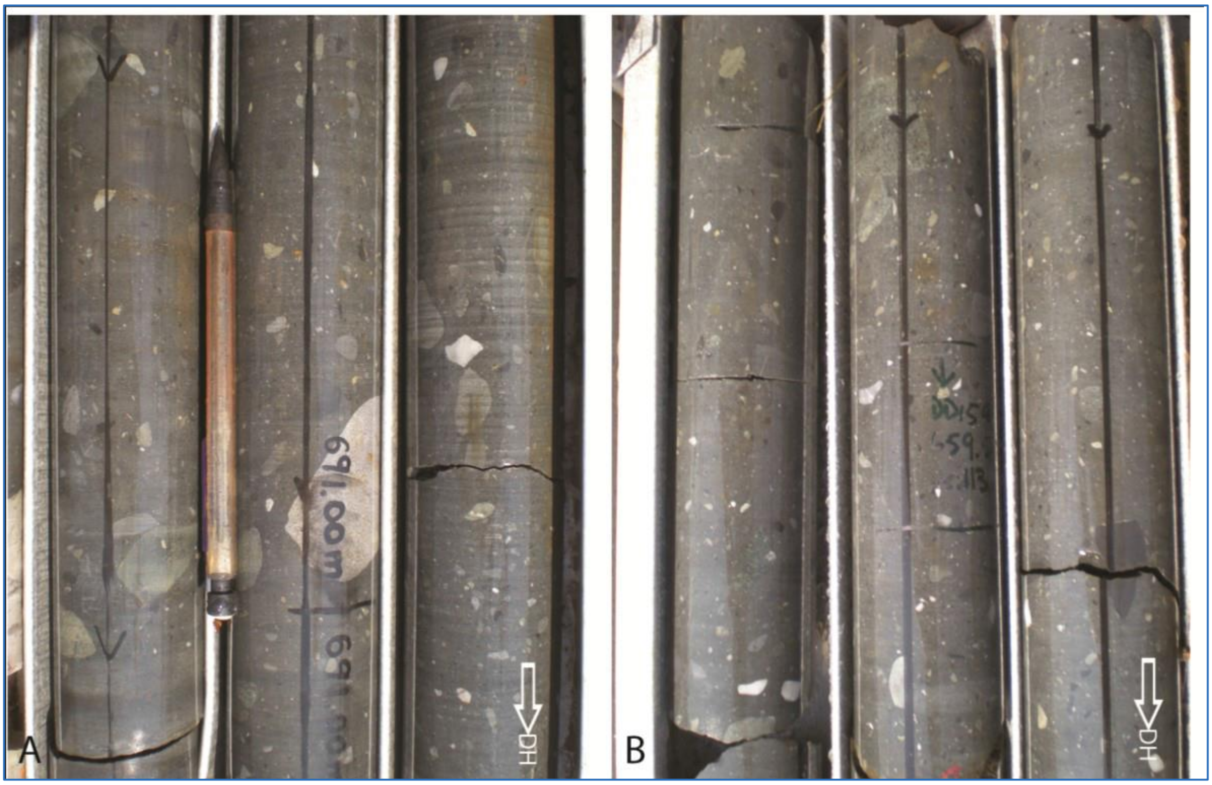


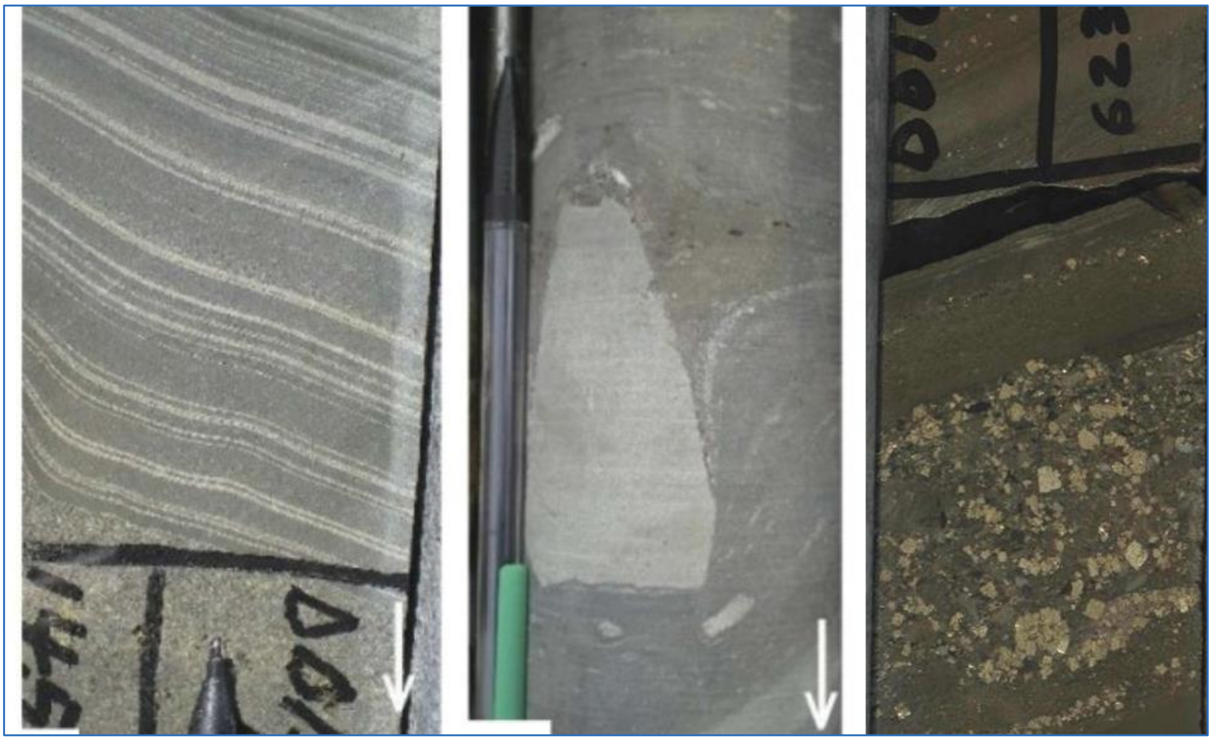
Figure provided by Ivanhoe, 2016

Figure 7.6 Clast-rich Diamictite (A) and Clast-poor Diamictite (B) from DKMC_DD159



Source: Schmandt et al (2013)

Figure 7.7 Distinctive Varves, Dropstone and Pyrite at the Base of the KPS in DKMC_DD154



Source Schmandt et al (2013); left picture shows varves in diamictite; centre picture shows dropstone deforming diamictite below; right picture shows coarse pyrite concordant with bedding.

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 7.8 through Figure 7.10), indicate a north–west stratigraphic thickness trend 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 7.11.

The intermediate siltstone (Ki1.1.1.2) shown in Figure 7.12 is either absent or locally developed as a single siltstone unit separating the clast-poor and clast-rich diamicties. 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, refer toFigure 7.13. 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 7.8 Ki1.1.1.1 Vertical Thickness

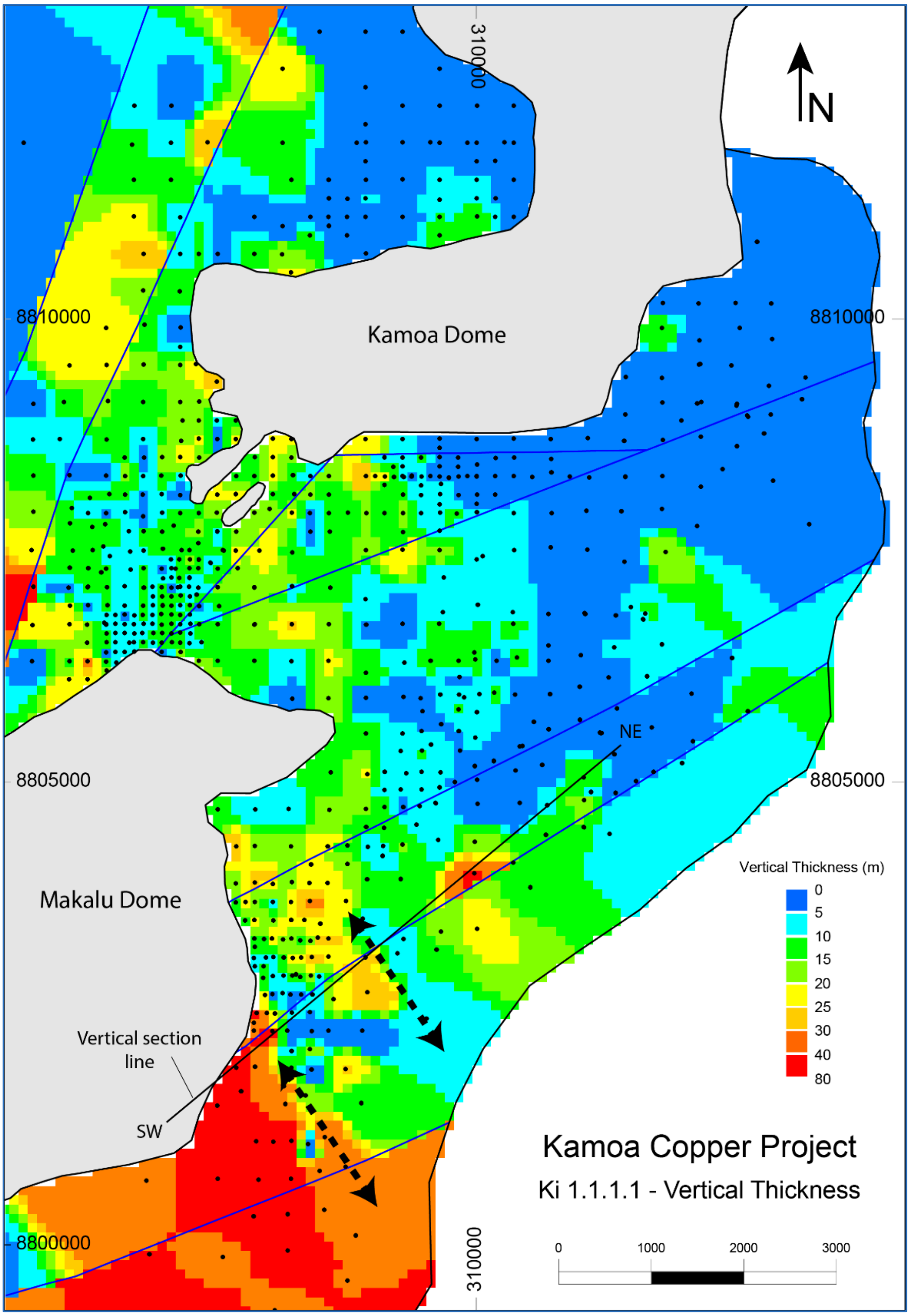


Figure provided by Ivanhoe, 2016; black line is cross section shown on Figure 7.11; grey areas represent domes or leached zones; blue lines are interpreted faults; black dashed lines are north–west thickness trend. Scale bar represents metres.

Figure 7.9 Ki1.1.1.3 Vertical Thickness

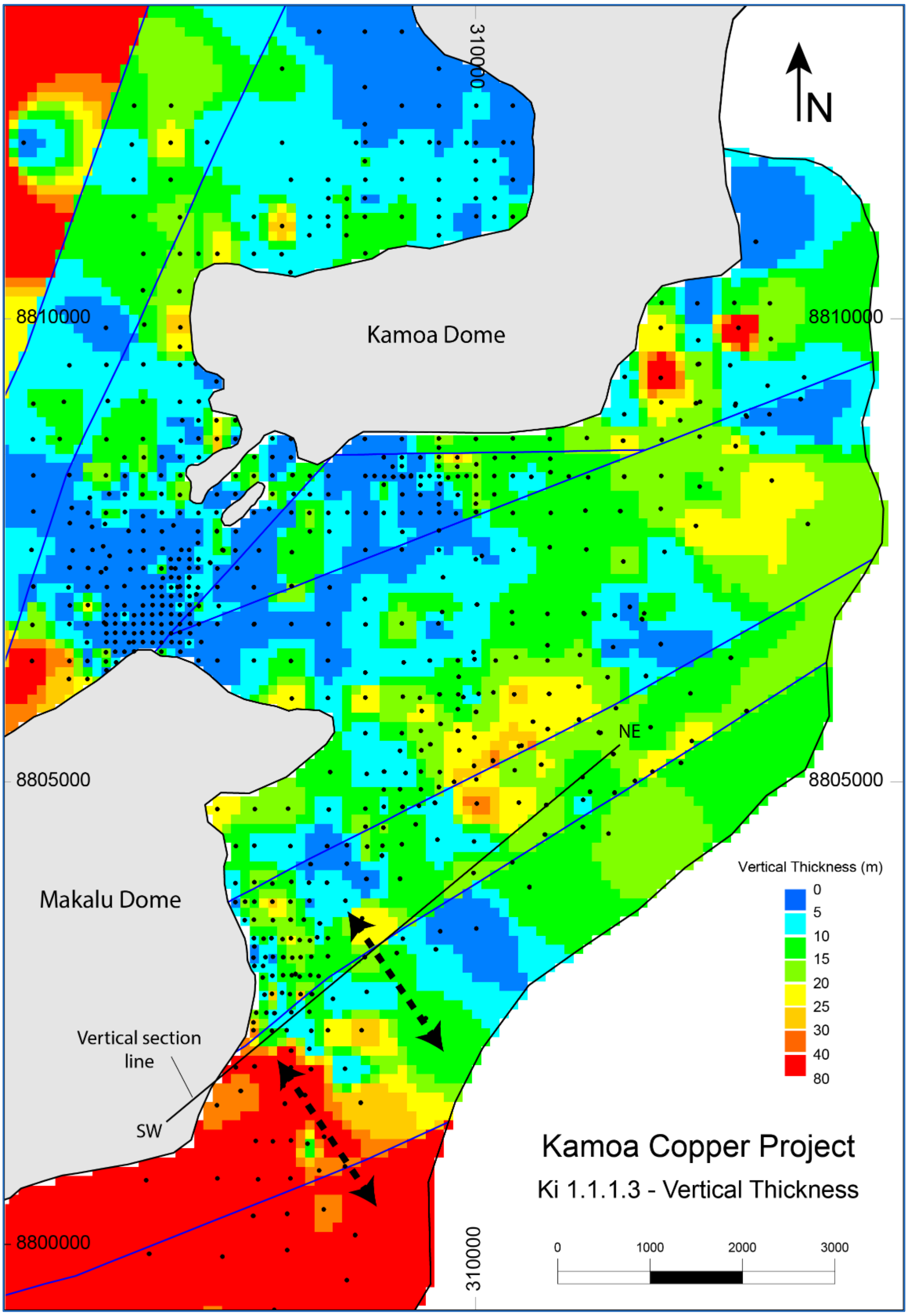


Figure provided by Ivanhoe, 2016; black line is cross section shown on Figure 7.11; grey areas represent domes or leached zones; blue lines are interpreted faults; dashed black line is north–west thickness trend. Scale bar represents metres.

Figure 7.10 Ki 1.1.2 Vertical Thickness (KPS)

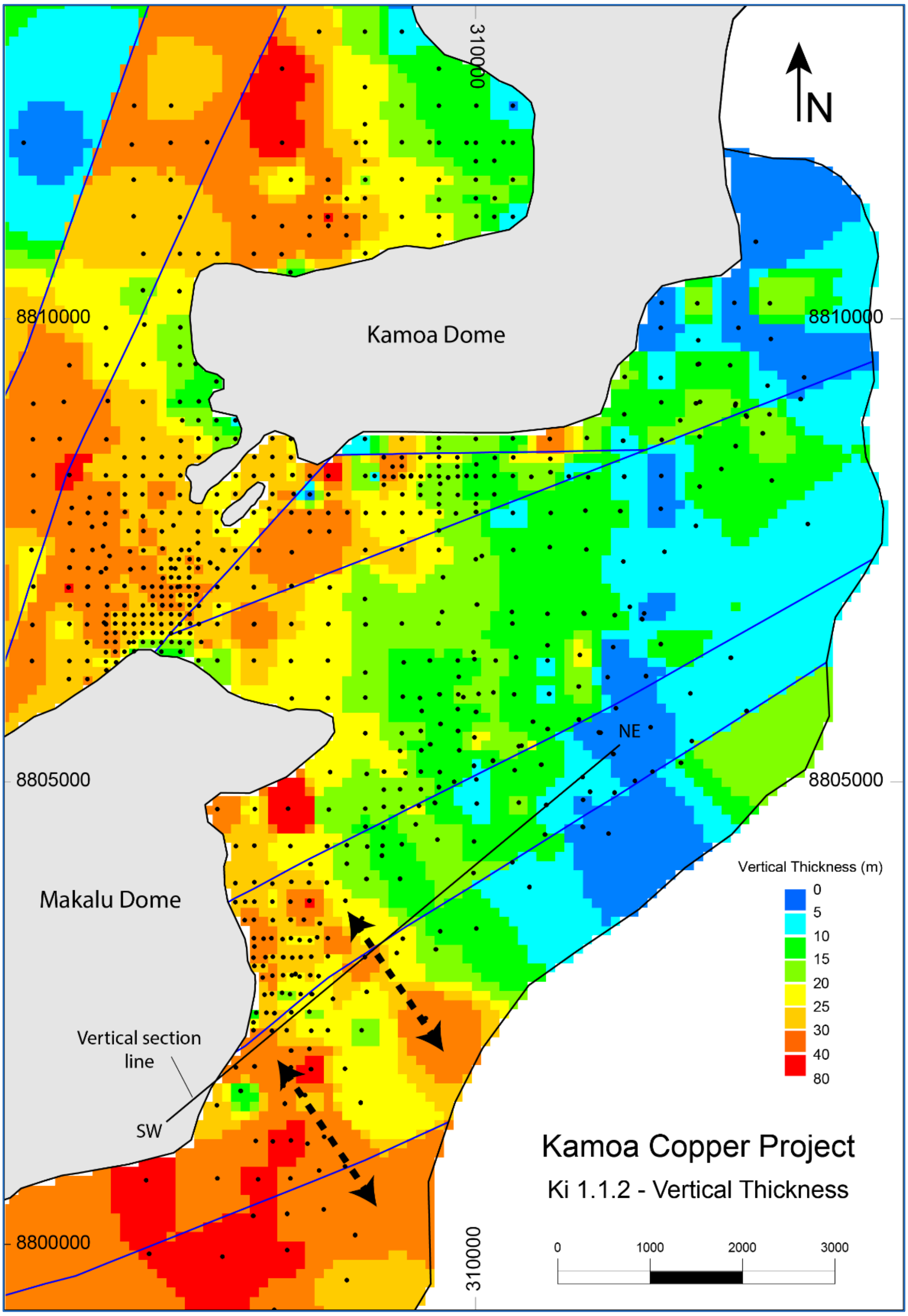


Figure provided by Ivanhoe, 2016; black line is cross section shown on Figure 7.11; grey areas represent domes or leached zones; blue lines are interpreted faults; dashed black line is north–west thickness trend. Scale bar represents metres.

Figure 7.11 Section from Kansoko Sud (SW) to Kansoko Centrale (NW)

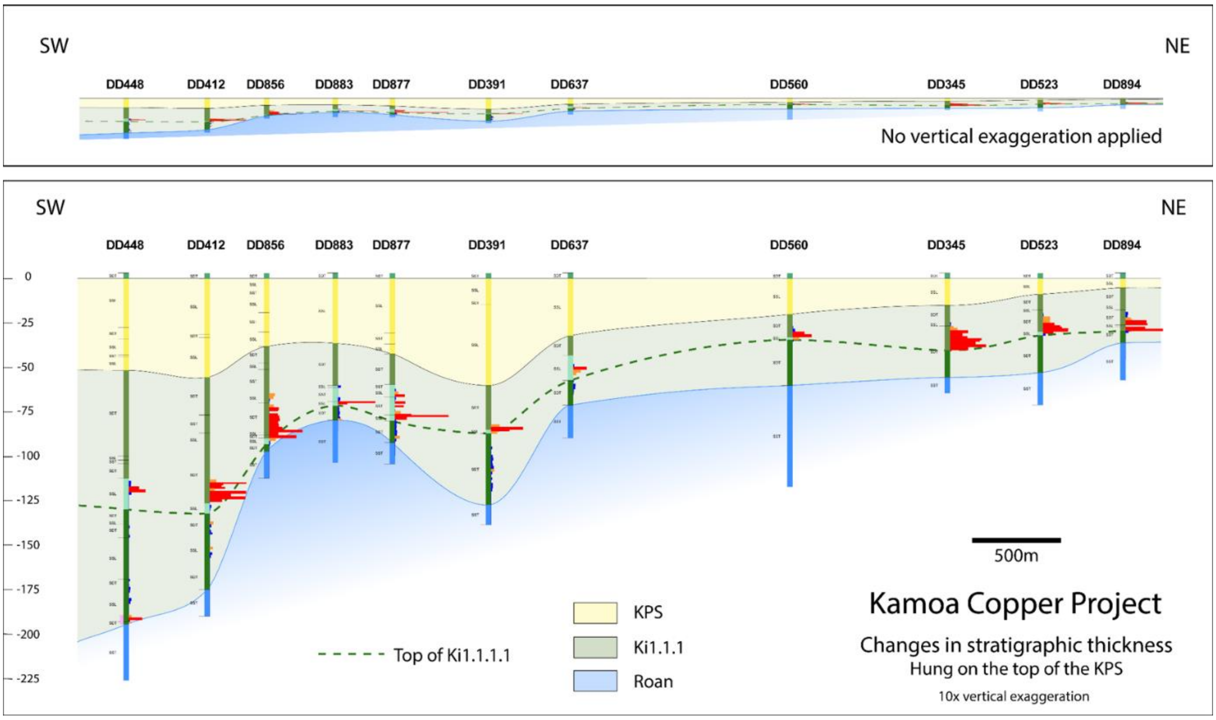


Figure provided by Ivanhoe, 2014, illustrating the thickening of units to the south–west; section line indicated on Figure 7.8 to Figure 7.10. Copper grades in percent, shown as red histograms.

Figure 7.12 Ki 1.1.1.2 Vertical Thickness (KPS)

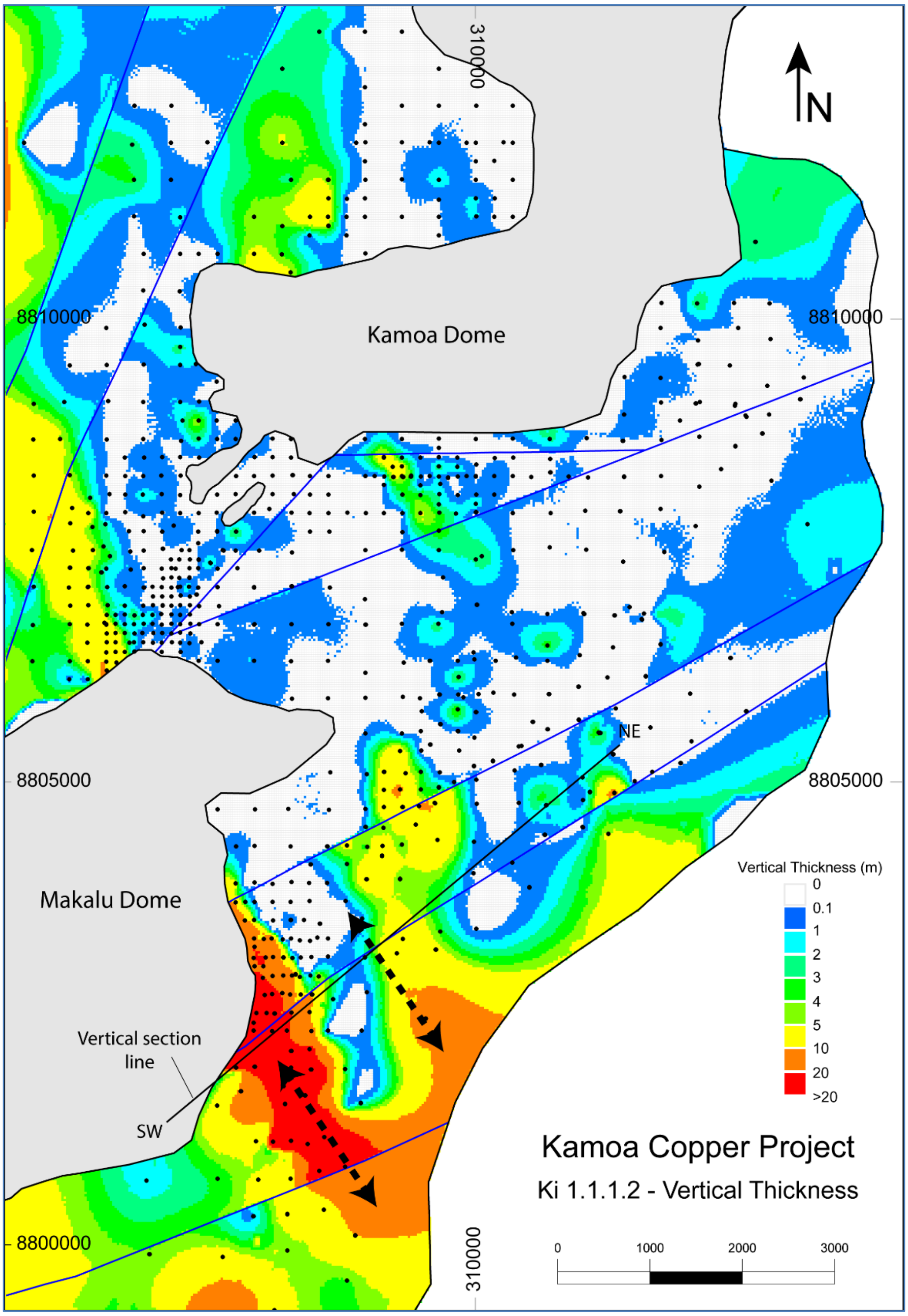


Figure provided by Ivanhoe, 2016; black line is cross section shown on Figure 7.11; grey areas represent domes or leached zones; blue lines are interpreted faults; dashed black line is north–west thickness trend. Scale bar represents metres.

Figure 7.13 Occurrence of Intermediate Siltstone Units within the Ki1.1.1

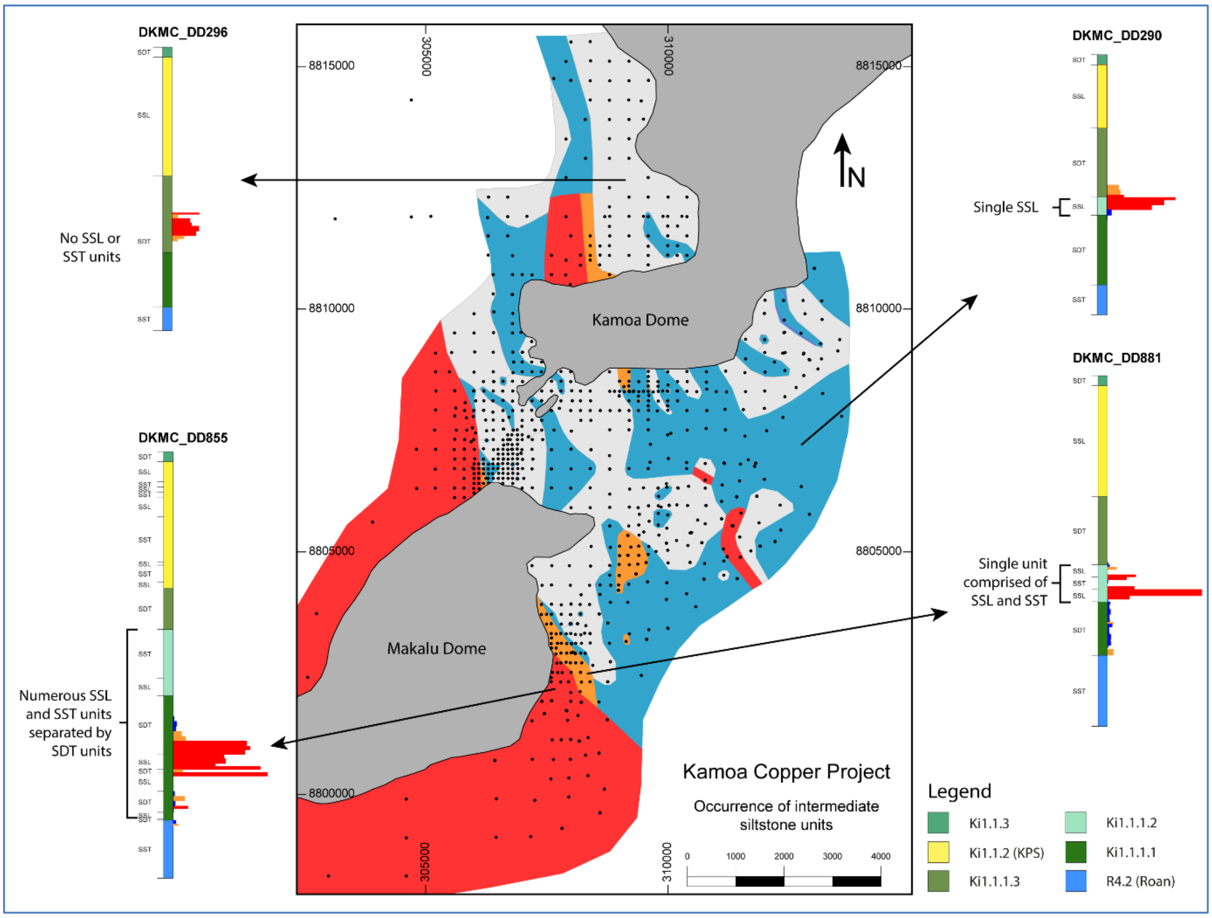


Figure provided by Ivanhoe, 2016, Scale bar represents metres grey areas represent domes or leached zones. Copper grades in percent, shown as red histograms. (SDT diamicite), (SSL siltstone), (SST sandstone)

7.3.3 Igneous Rocks

Andesite/dolerite sills and vesicular to massive lava have been identified in the Project area:

- Andesite/dolerite sills or flows: These occur as one or more, 5 m to 80 m thick, apparently concordant tabular bodies in the northern end of the Kansoko Trend.
- Vesicular to massive lava: Drillhole DKMC_DDD032 in Exploration Permit 704 penetrated >200 m of vesicular to massive and brecciated and altered, pale green to light grey–green lava grading into a fine-grained igneous rock, probably of andesitic composition.

7.3.4 Structure

SRK (Vancouver) have continued to work on the interpretation of the structural model. Geophysical data (primarily the magnetics first vertical derivative) provide the primary support for continuity of structural features, whilst the drillhole data (up to August 2013), geotechnical logging and topographic lineaments all provide supporting evidence (refer to Figure 7.14).

Ongoing interpretation has sought to develop a broader structural framework for Kamoa. The 2014 structural model 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 resource model. These structures were used as boundaries to divide the mineralisation into structural zones, refer to Figure 7.15.

Figure 7.14 Structural Model Overlaid on Second Order Vertical Derivative Magnetic Image

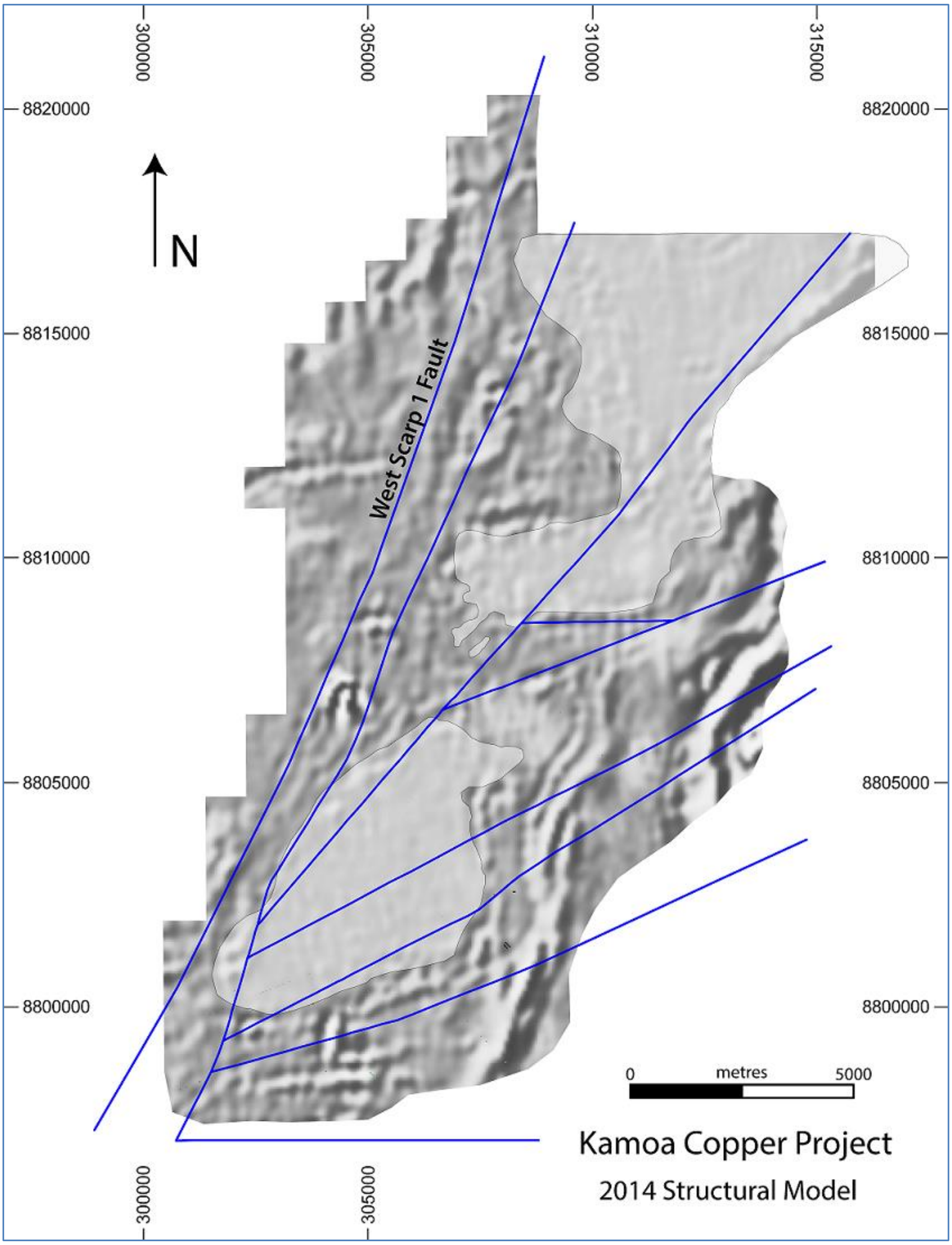


Figure proved by Ivanhoe, 2016; blue lines are interpreted fault traces; light grey areas are domes and surrounding leached zones.

Figure 7.15 Structural Model and SMZ Contours (masl)

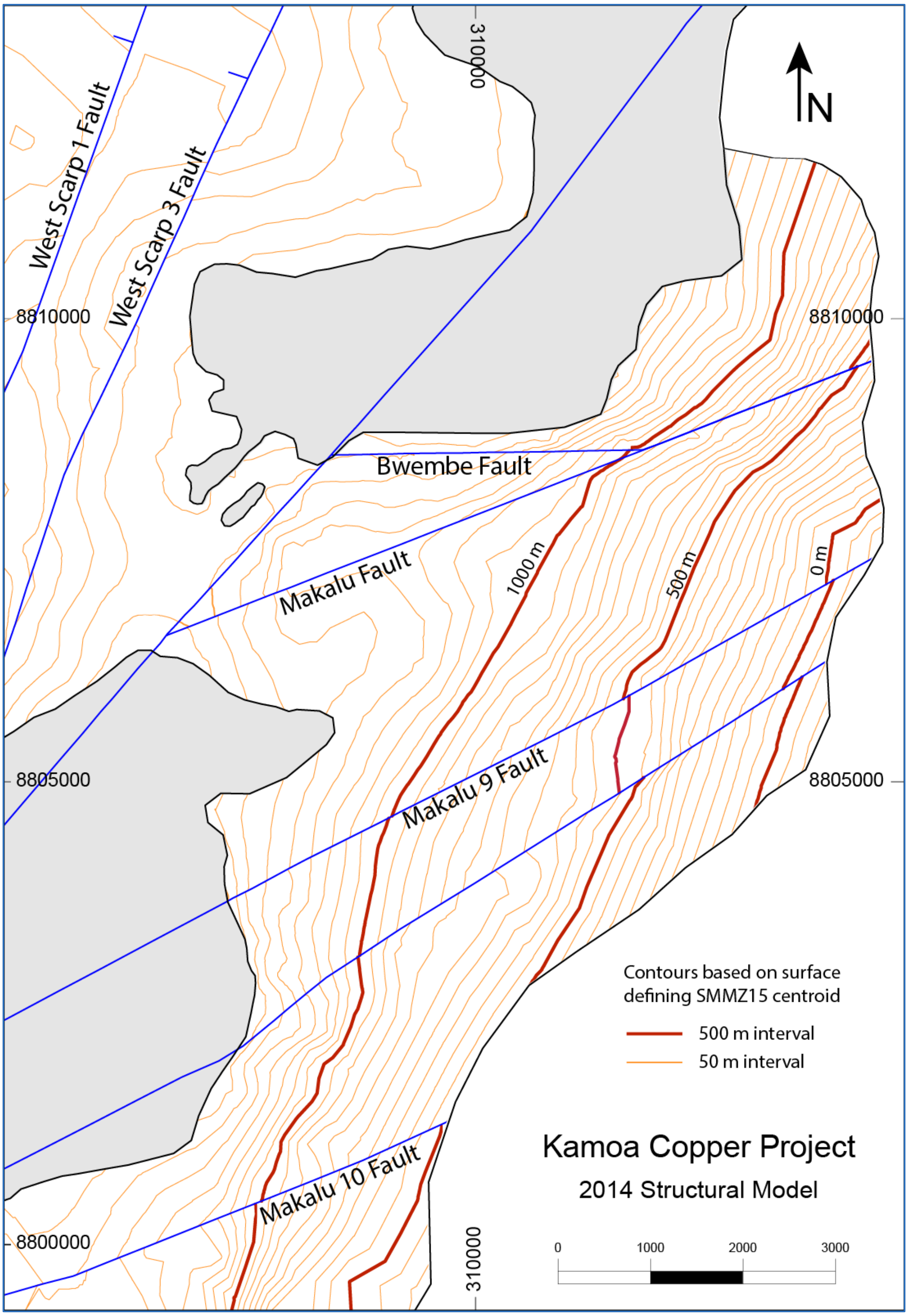


Figure provided by Ivanhoe, 2016; grey areas are domes and surrounding leached zones. Scale bar represents metres.

The presence of very open folds at Kamoa has been highlighted during the ongoing investigations. These folds 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 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-oriented normal faults, which truncate the western edge of the area. The West Scarp 1 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 7.16.

Figure 7.16 Structural Influences on Topography

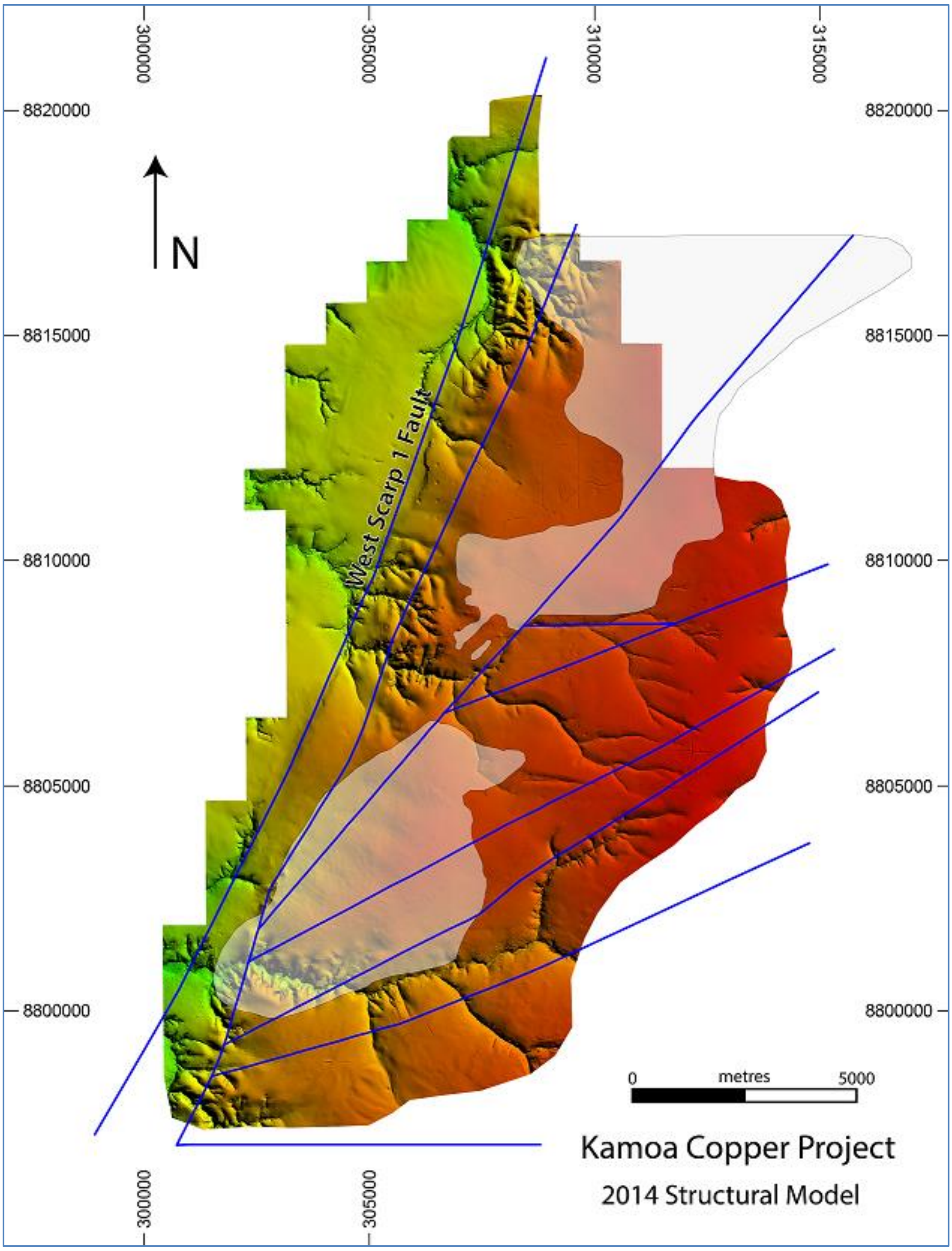


Figure provided by Ivanhoe, 2016; blue lines are fault traces; light coloured areas are domes and surrounding leached zones.

The stratigraphic units generally dip gently away from the dome edges at between 5° to 20°. Kamoa and Kamoa Nord areas are particularly gentle-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°.

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 hematite. 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 1 Fault.

7.3.7 Mineralisation

Mineralization at the Kamoa Project has been defined over an irregularly–shaped area of 20 km x 15 km. Mineralization is typically stratiform, and vertically zoned from the base upward with chalcocite (Cu_2S), bornite (Cu_5,FeS_4) and chalcopyrite ($CuFeS_2$). There is significant pyrite mineralization above the mineralized 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 mineralized body ranges from 0° to 10° near–surface above the Kamoa dome, to 15° to 20° on the flanks of the dome. Mineralization thicknesses at a 1.0% Cu cut–off grade range 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 east and south.

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 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 Kolwesi deposits, mineralisation at Kamoa is characterized by a lack of cobalt (Schmandt et al, 2013). Very little oxide mineralisation is evident at Kamoa, and likely due to the leaching effects 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, refer to Figure 7.17. Relatively laterally narrow zones of supergene enrichment are also observed in these areas; however, the bulk of the copper mineralisation is hypogene.

Figure 7.17 Schematic Illustrating the Transition from Leached to Supergene to Hypogene Mineralisation in Kamoa Sud

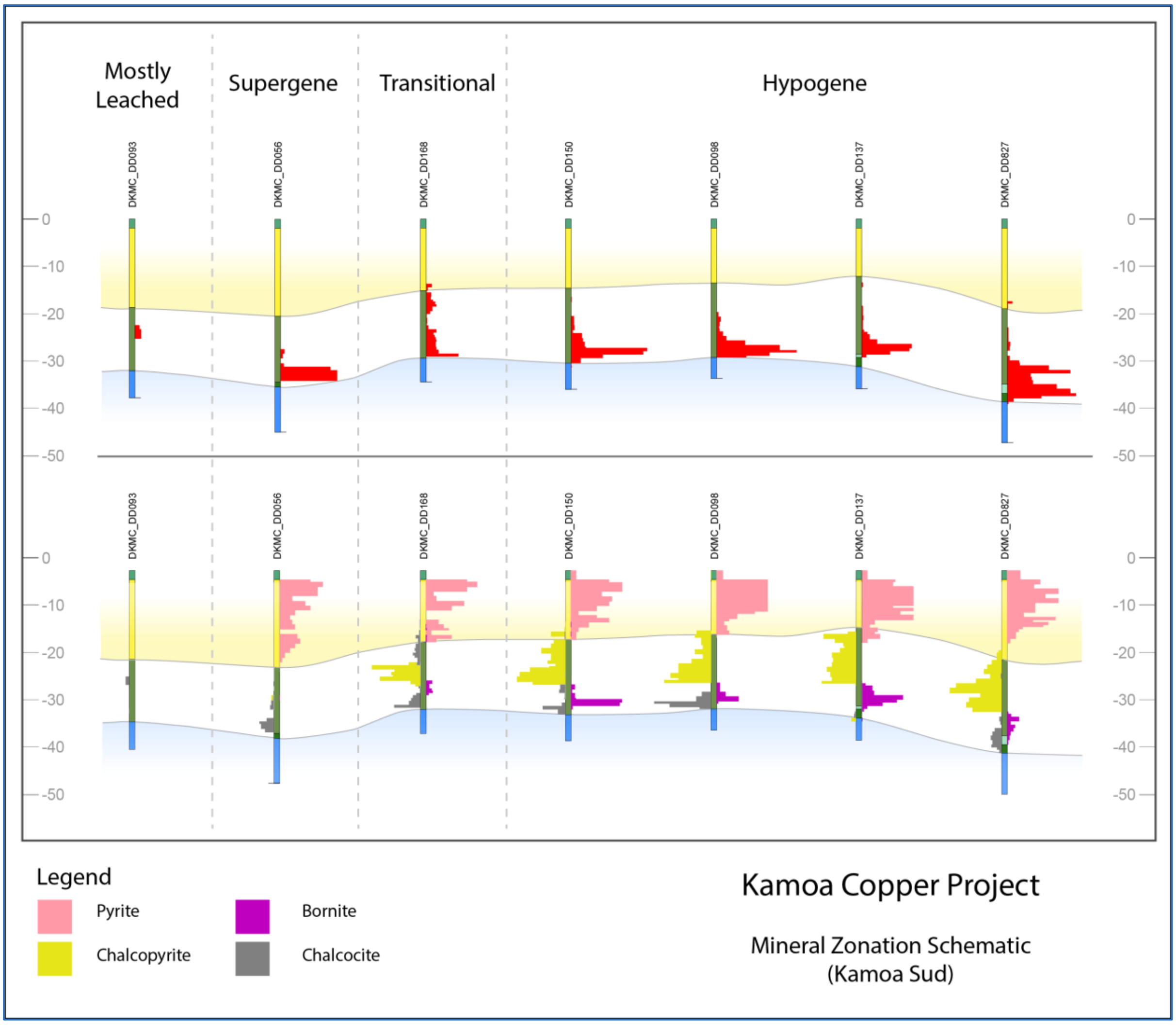


Figure provided by Ivanhoe, 2014; the upper panel displays copper grades; the lower panel displays sulphide species abundance. Copper grades in percent, shown as red histograms.

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. On a local scale, the clast-rich diamictite (Ki1.1.1.1) is considered to be only weakly reducing, and thus generally hosts only low-grade (<0.5% Cu) 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 most of the primary mineralised zone. Some of the most consistent and highest grade intervals are intersected where the clast-rich diamictite is absent, and the clast-poor diamictite rests directly on the Roan contact.

The earliest sulphide mineralisation at Kamoa 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 (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 SMZ is presented as a lateral mineral zonation, refer to Figure 7.18. The change from supergene to hypogene is generally transitional with a strongly developed vertical zonation evident in the hypogene (refer to Figure 7.19).

Figure 7.18 Lateral Extent of Sulphide Species Zonation at Kamoa

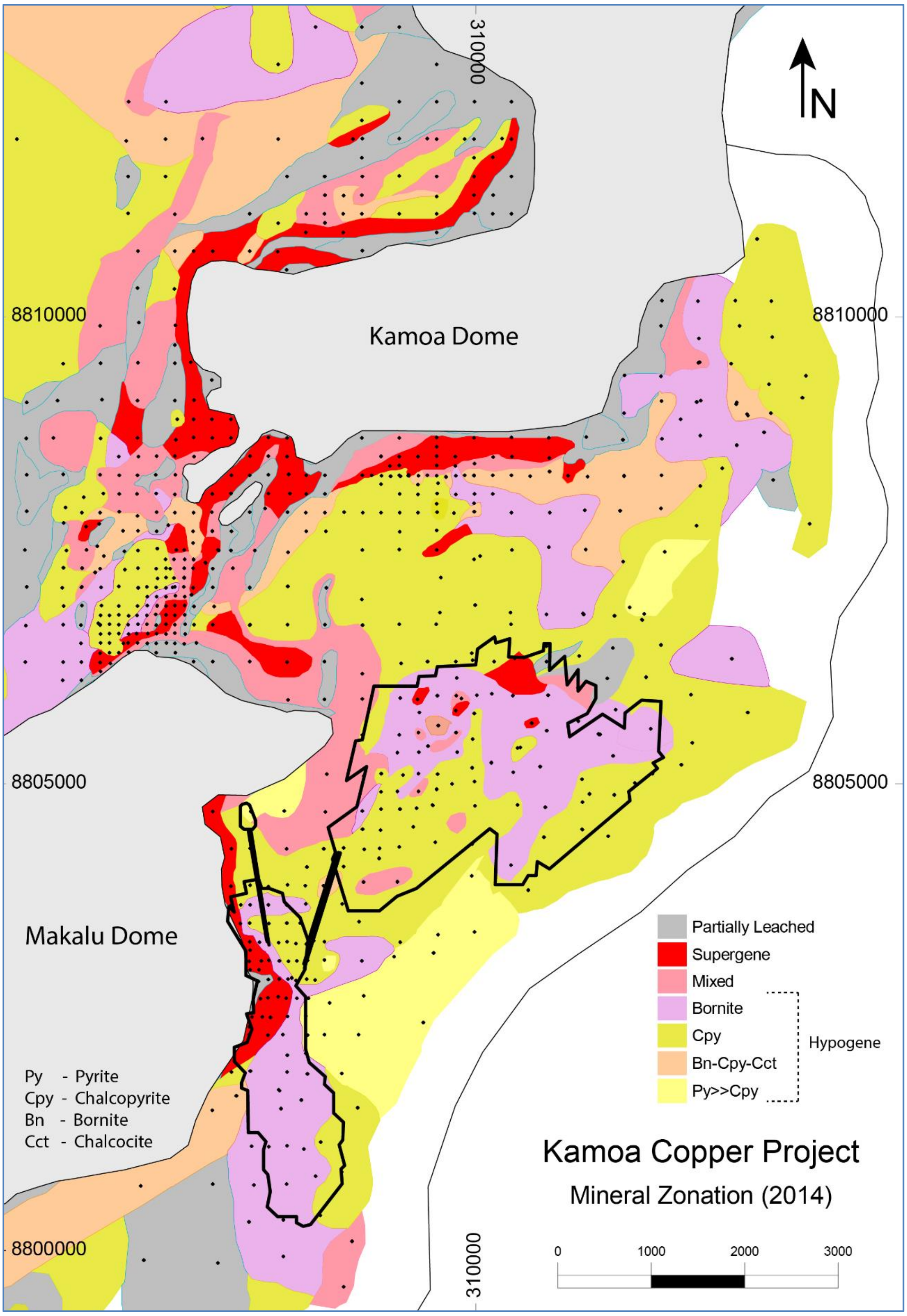


Figure proved by Ivanhoe, 2016. Dark black line is the area covered by the Phase 1 mine plan. Scale bar represents metres.

Figure 7.19 Schematic of Mineral Zonation at Kamoa

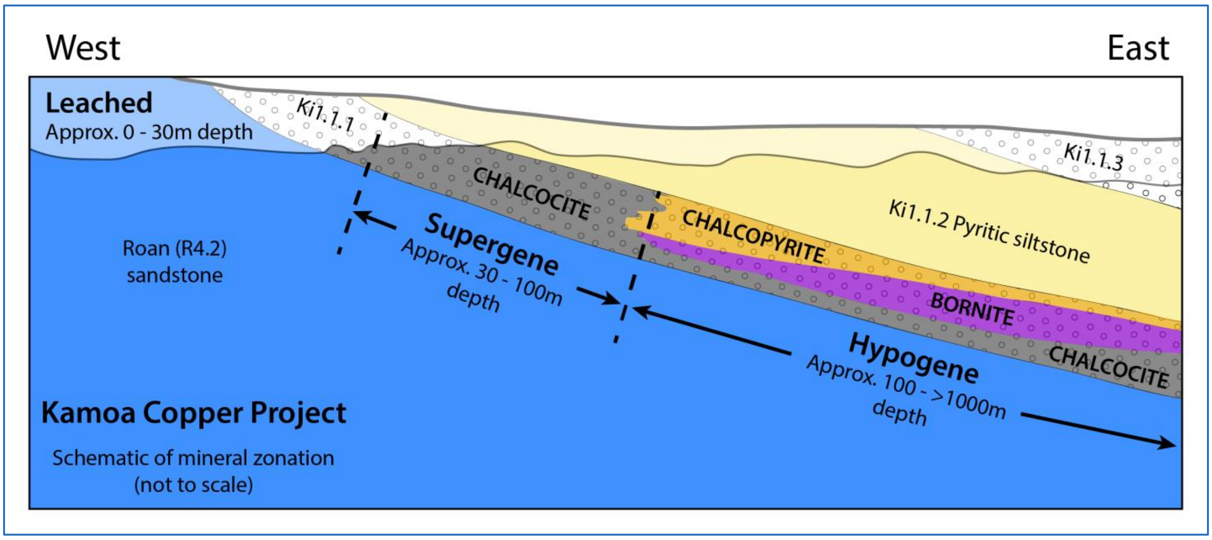


Figure is schematic and not to scale. The true thicknesses of drilled composites range from 2.7–18 m with an average of 5.6 m. Leached zone ranges from 0 to 30 m vertical depth from surface. Supergene zone ranges from 30 m to 100 m vertical depth from surface. Hypogene typically extends from 100 m vertical 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, refer to Figure 7.20. The supergene zone may extend to depths of 250 m or more along fracture zones and stratigraphic contacts (Schmandt et al, 2013).

Hypogene copper mineralisation at Kamoa exhibits a distinct vertical zonation. In well-mineralised intercepts the full zonation is evident, with chalcocite and/or bornite occurring at the base of the mineralised interval (often best developed where the mineralised zone occurs directly on top of the Roan). This zone grades stratigraphically upward into a chalcopyrite-rich zone, refer to Figure 7.21. The zonation is capped by pyrite-rich siltstones. Where the mineral zones overlap, the more copper-enriched sulphides are observed to rim or replace the more copper-poor sulphides (Schmandt et al, 2013).

Chalcocite and bornite are typically very fine grained (<50 µm), and are sometimes only identified visually from the change in colour evident in the matrix of the rock. Chalcopyrite is typically coarser grained and can form very distinct rims to clasts within the diamictite, occurring predominantly on the top of clasts (Schmandt et al, 2013). 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 7.22 illustrates the moderate foliation defined by the alignment of fine and coarse-grained chalcopyrite and the development of an asymmetric up-hole strain shadow parallel to the foliation in DKMC_DD909.

Within the strata in a drillhole, fine-grained and coarse-grained sulphide assemblages are generally similar, suggesting they are coeval (Schmandt et al, 2013).

Figure 7.20 Examples of Supergene Chalcocite

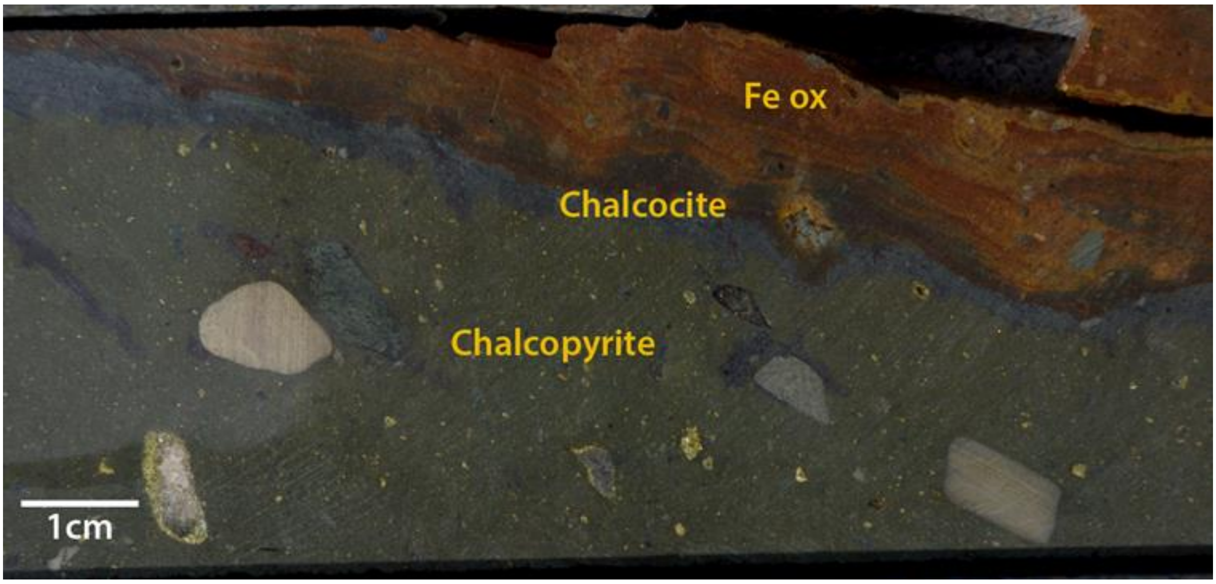
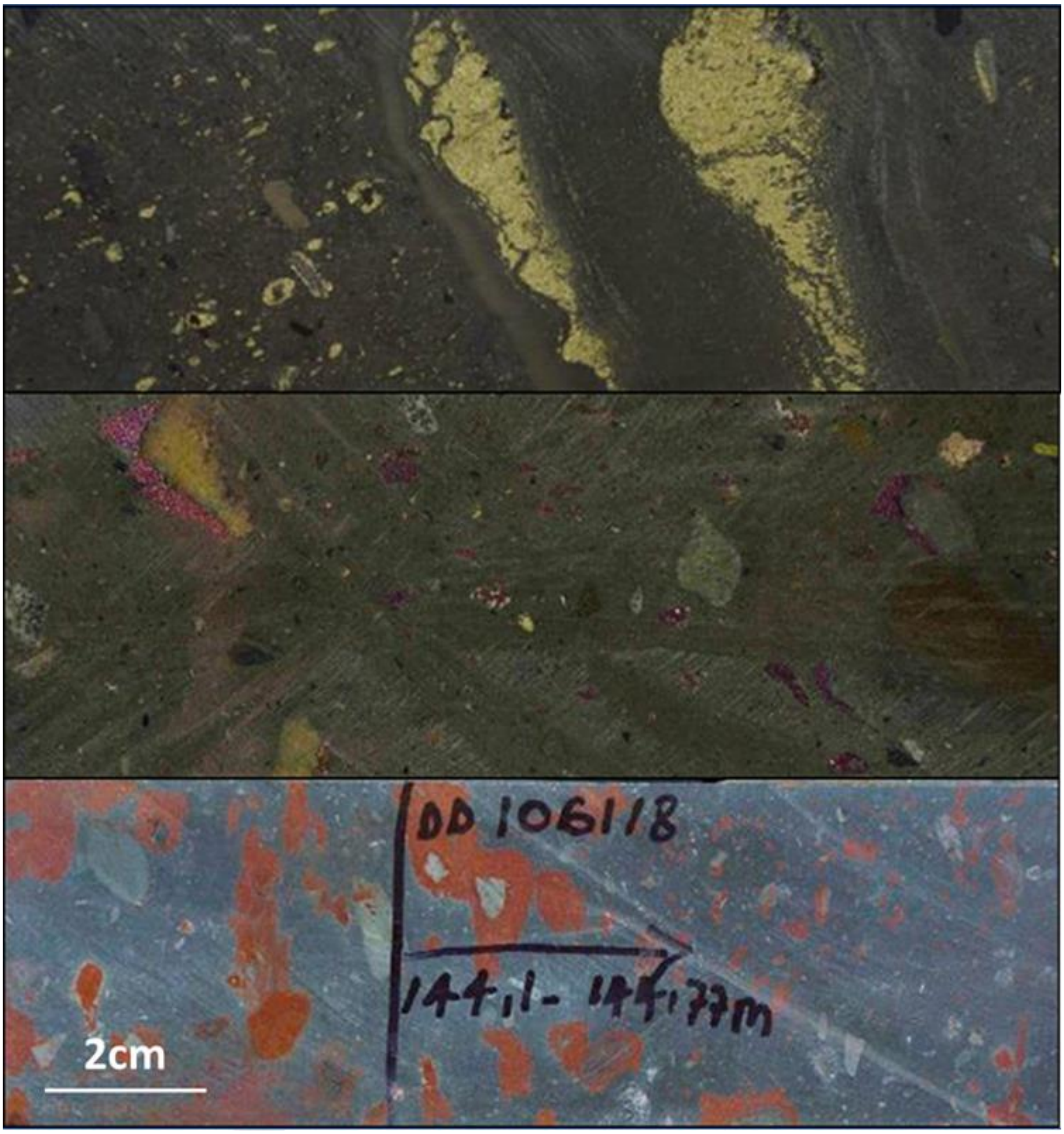


Figure by Ivanhoe, 2016.

Figure 7.21 Examples of Hypogene Mineralisation



Coarse chalcopyrite (yellow, top), coarse- to fine-grained bornite (magenta, middle) and fine-grained chalcocite (silver, bottom). Figure by Ivanhoe, 2016.

Figure 7.22 Strain-shadow in DKMC_DD909

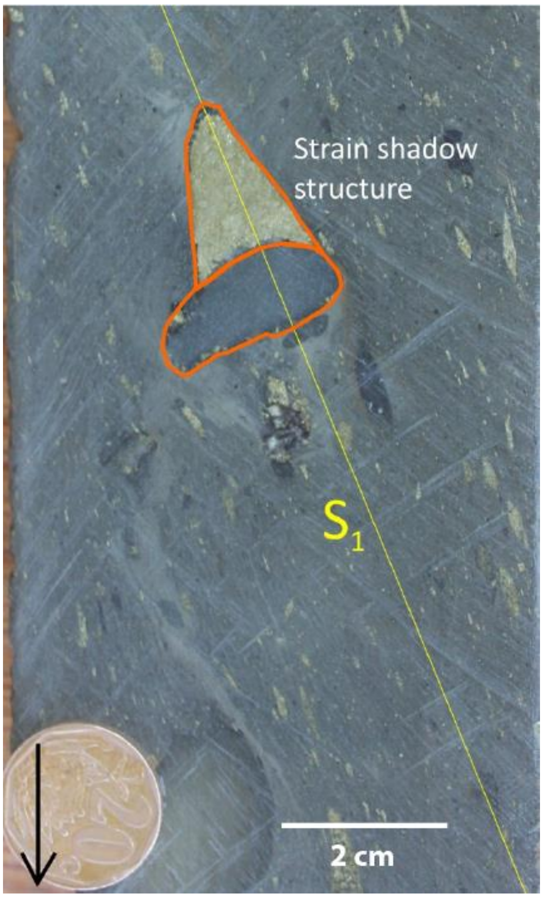


Figure by Ivanhoe, 2016.

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 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 7.23.

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-rich 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 7.24, with typical grade profiles for each zone illustrated in Figure 7.25.

Figure 7.23 Stratigraphic Section showing Continuity of Mineralisation near Base of Ki 1.1.1.3

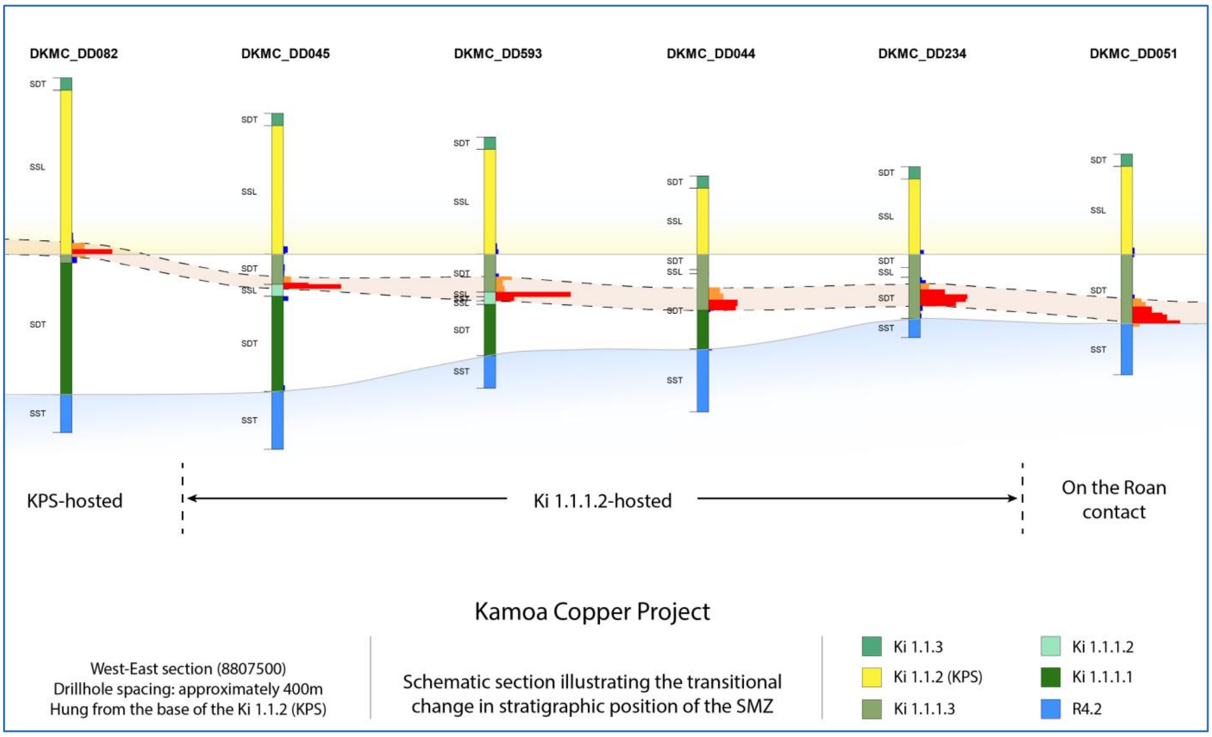


Figure provided by Ivanhoe, 2014. Copper grades in percent, shown as red histograms.

Figure 7.24 Facies in which Mineralisation Occurs

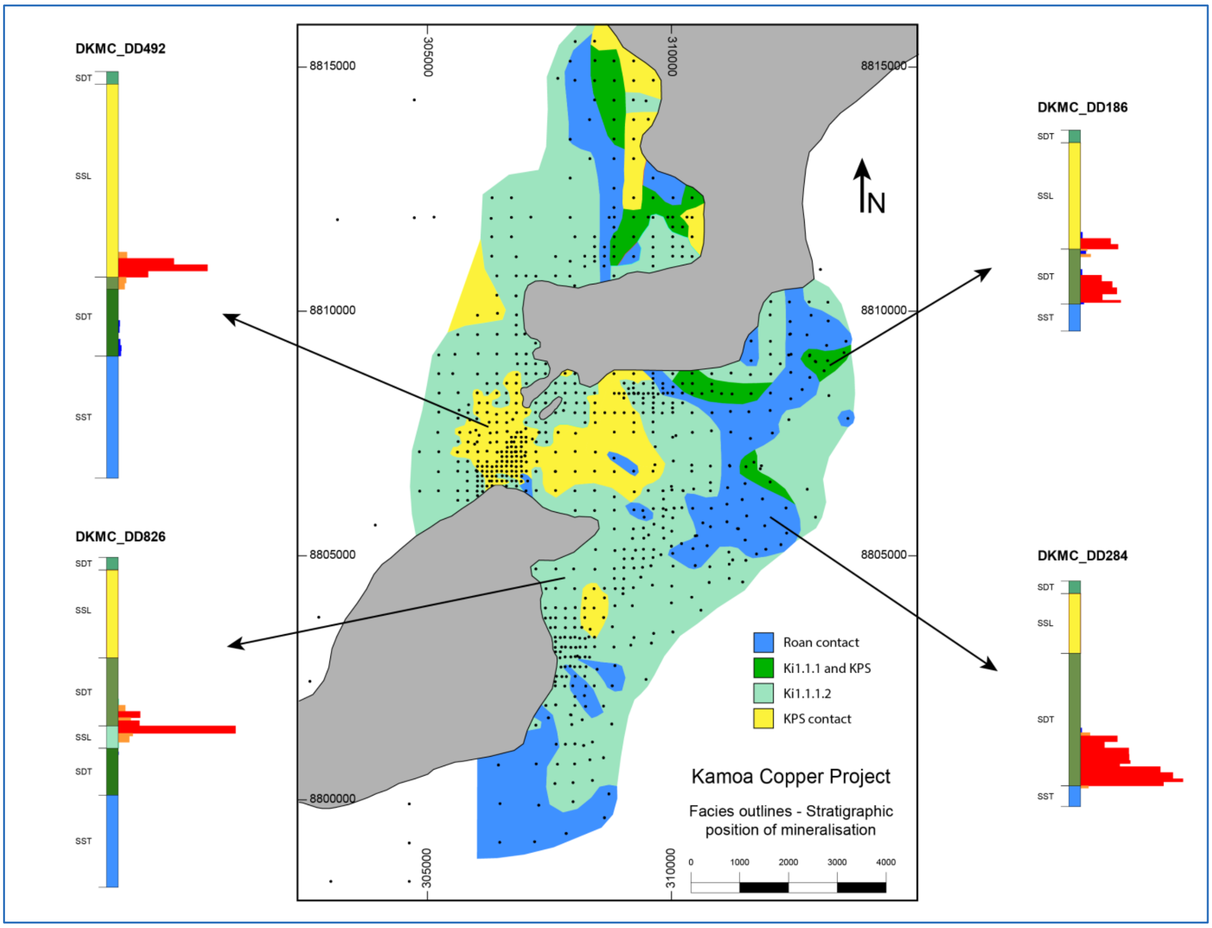


Figure provided by Ivanhoe, 2016. Scale bar represents metres. Copper grades in percent, shown as red histograms.

Figure 7.25 The Impact of Stratigraphy on the Characteristics of the Grade Profile

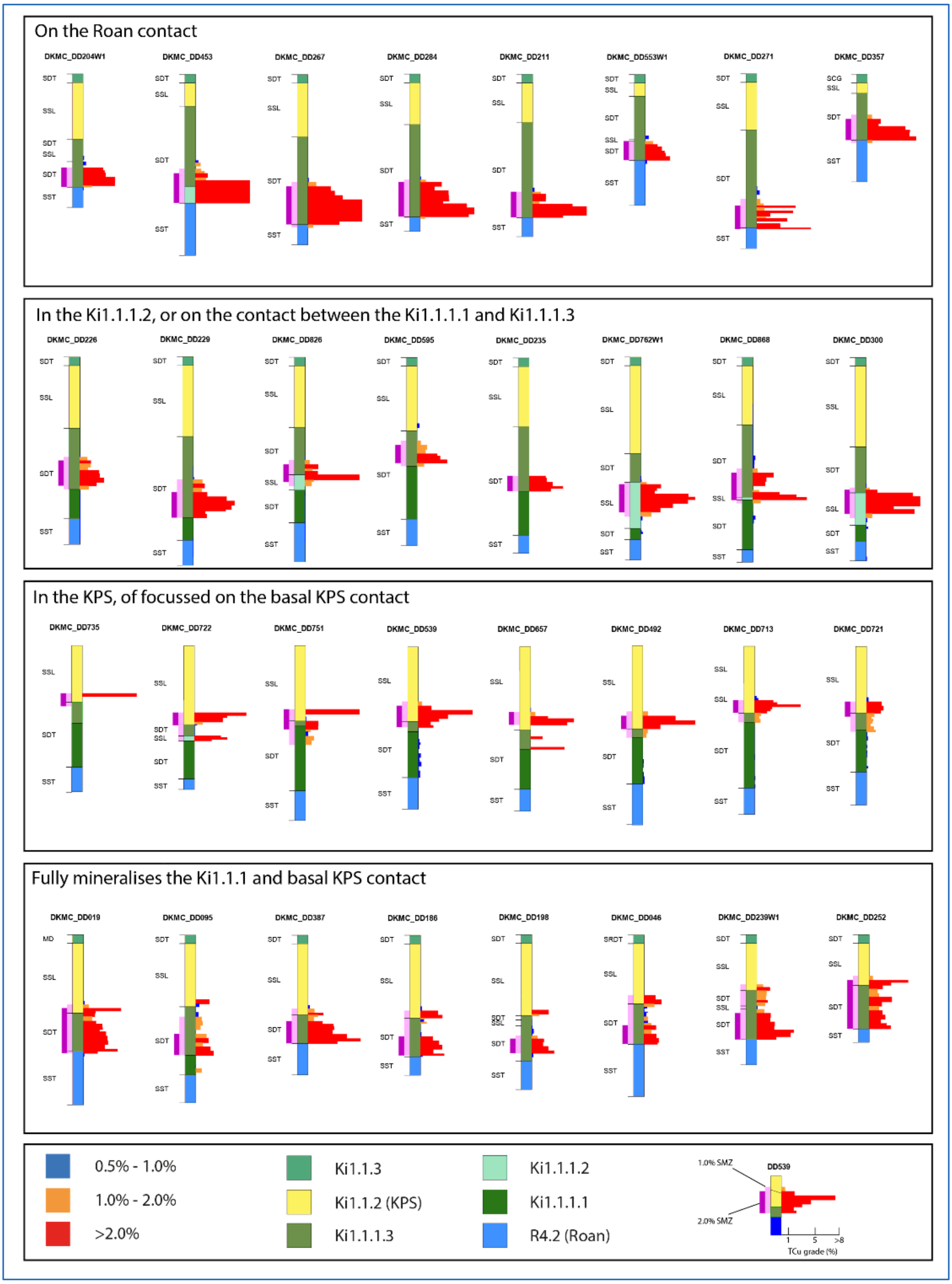


Figure provided by Ivanhoe, 2014. Drillholes are hung from the topo of the KPS.

7.3.8 Studies Relating to Controls on Mineralisation

A number of controls on mineralisation are evident. The primary control is the position of the reducing horizon, which at Kamoa is typically the base of the clast-poor (Ki1.1.1.3) diamictite. Weathering can also have a local control on mineralisation through the development of an enriched supergene zone. Two additional controls have been investigated and used in the estimation of Mineral Resources.

Indicator Variogram Study

Traditional variograms of TCuxTT (%) within the mineralised zone do not suggest a strong anisotropy to the grade. An indicator variography study using deciles of the TCuxTT (%) population for the mineralised zone did, however, identify two distinct orientations, refer to Figure 7.26.

The lower deciles exhibited a fairly strong north–east orientation, which aligns with the faults that are interpreted to have offset the stratigraphic sequence. Continuity directions change fairly quickly to align north–west at higher-grade deciles. This direction (320° to 330°) aligns with that observed from the north–west orientated syn-sedimentary structural controls evident in the vertical thickness plots of the individual stratigraphic units.

Figure 7.26 Variogram Fans at the 20th and 90th Percentiles Illustrating the Change in Continuity Direction

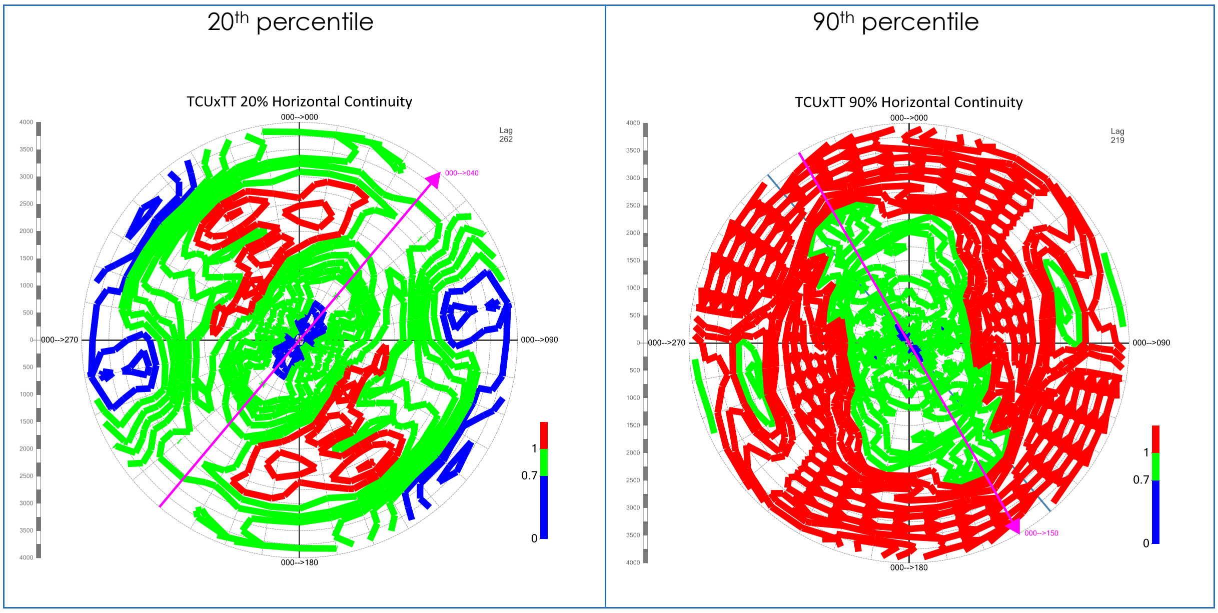


Figure provided by Ivanhoe, 2014.

Selective Mineralised Zone Sensitivity Study

The stratigraphic position of the mineralisation exerts a strong control on the nature of the grade profile, which directly impacts the definition of the selective mineralised zone (SMZ) during resource modelling. The Mineral Resource has traditionally been defined through the selection of a SMZ at a 1.0% total copper (TCu) cut-off, refer to Figure 7.27. Sensitivity testing was done at progressively higher cut-offs (in 0.5% TCu increments) to determine how the vertical position of the mineralised zone would change at higher cut-offs. A set of SMZs was redefined for each cut-off, and these sets are nested within the base case SMZ (1% TCu cut-off). The position of (metres away from) the higher cut-off SMZ relative to the footwall and hangingwall position of the 1.0% TCu cut-off was determined. Figure 7.28 shows the results for a 2.0% TCU cut-off.

The dominant style of mineralisation at Kamoa is Bottom loaded, particularly in areas where the SMZ is located at the base of of Ki.1.1.1.2 or Ki1.1.1.3. If the SMZ cut-off is increased, the thickness of the SMZ decreases. In the majority of cases (Bottom loaded), the most significant reduction is relative to the hangingwall position; i.e. the SMZ becomes narrower at higher cut-offs by removing lower-grade material primarily from the hangingwall side. The exception to this is where the mineralisation is partially located in the KPS and partially in the underlying Ki1.1.1 diamictite. In these cases, the KPS mineralisation tends to be higher grade; thus the higher cut-off SMZs retain the mineralisation in the KPS, but exclude the footwall mineralization in the Ki1.1.1.

Figure 7.27 The Selection of the Selective Mineralised Zone (SMZ) at a 1% TCu cut-off

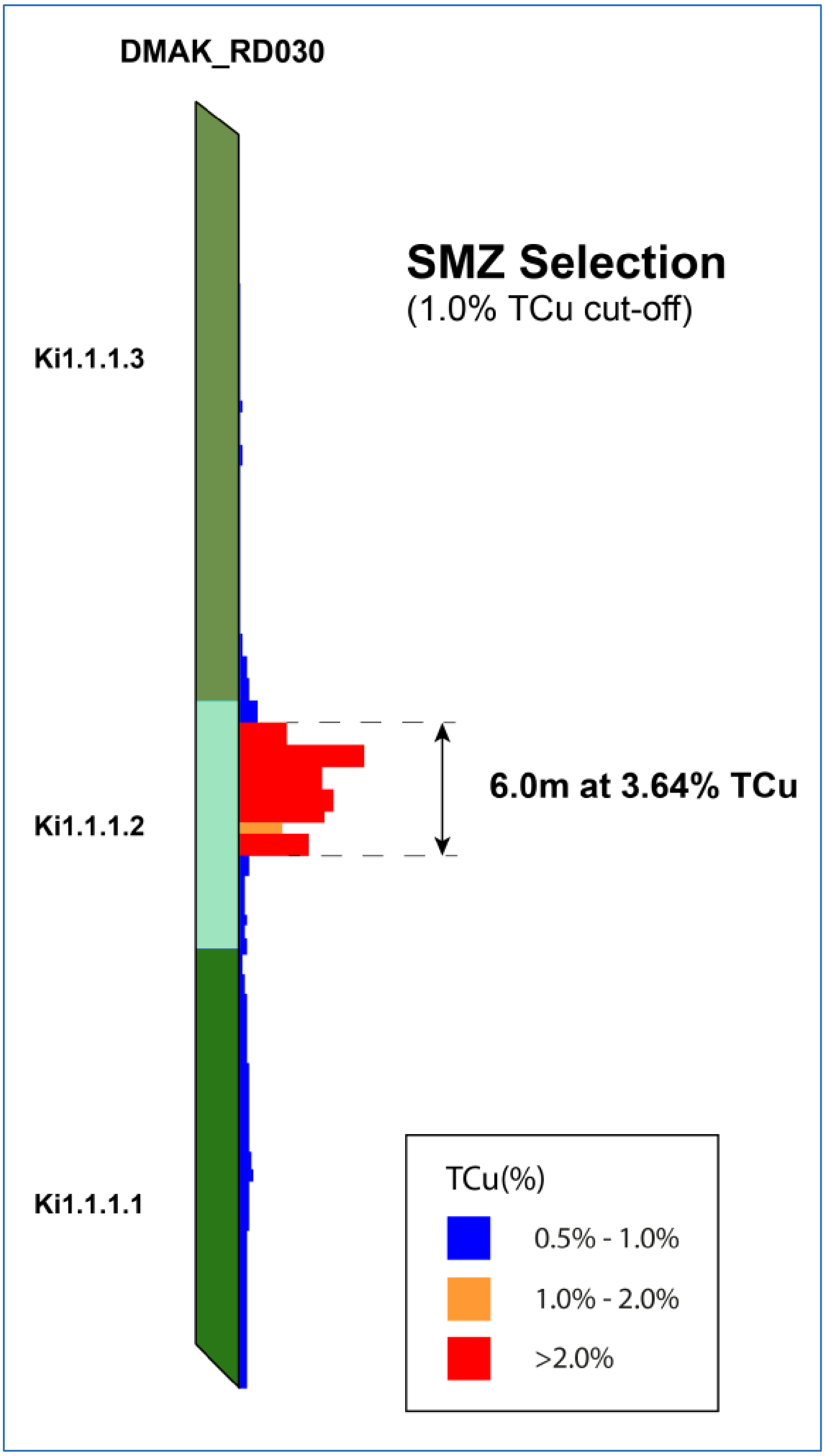


Figure provided by Ivanhoe, 2016.

Figure 7.28 Position of the 2.0% TCu Cut-off SMZ at Incrementally Higher Cut-off Grades Relative to the Footwall (left) and Hangingwall (right) of the SMZ at a 1.0% TCu Cut-off

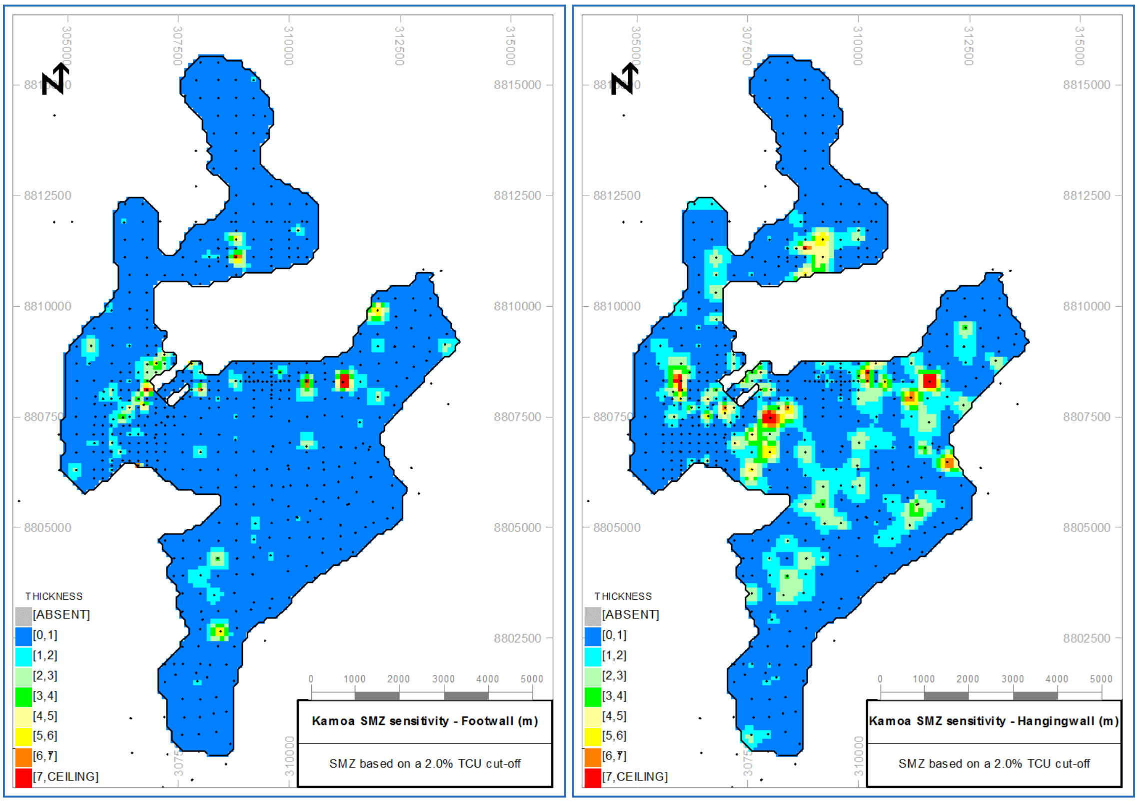


Figure provided by Ivanhoe, 2014. Scale bar represents metres.

Mosaic Patterns to Mineralization

Foreland-hosted copper deposits such as Kamoa 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 7.29 shows the 2014 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 7.29 2014 Mineral Resource Model for TCu Grade and True Thickness

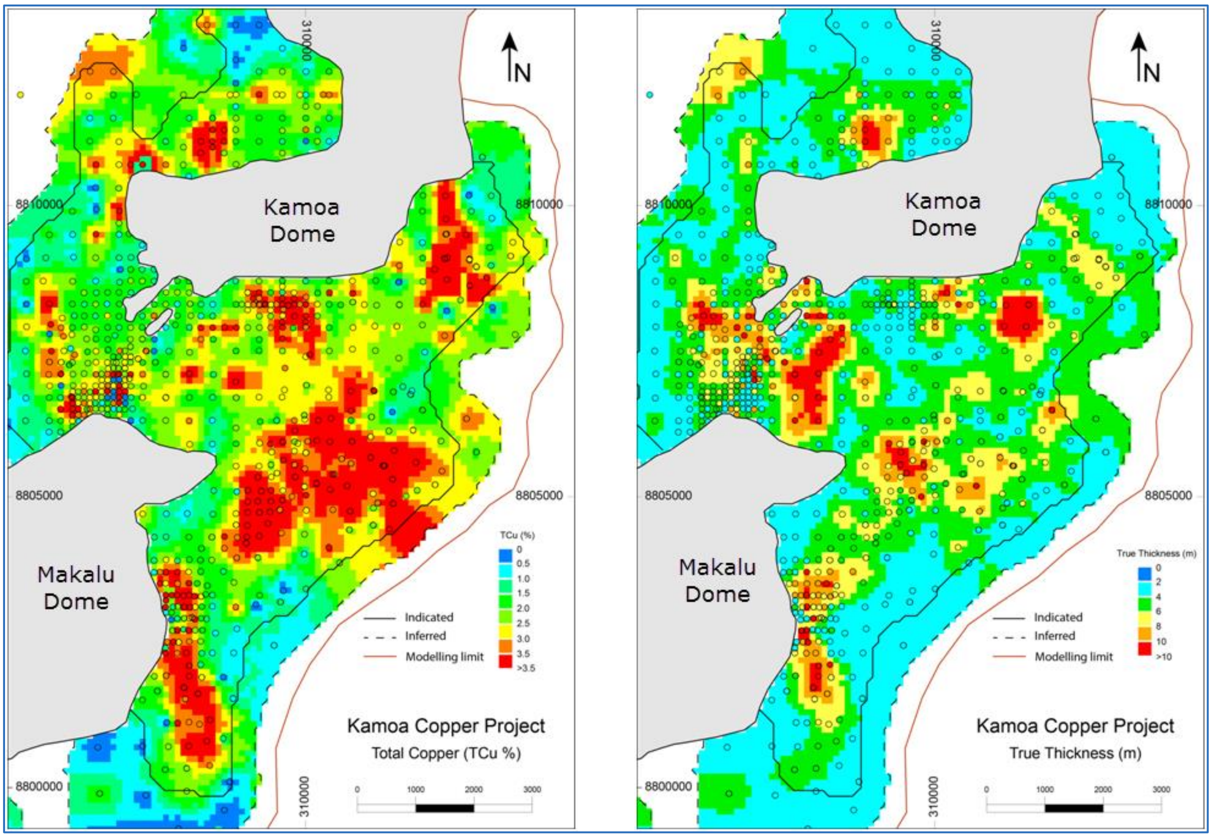
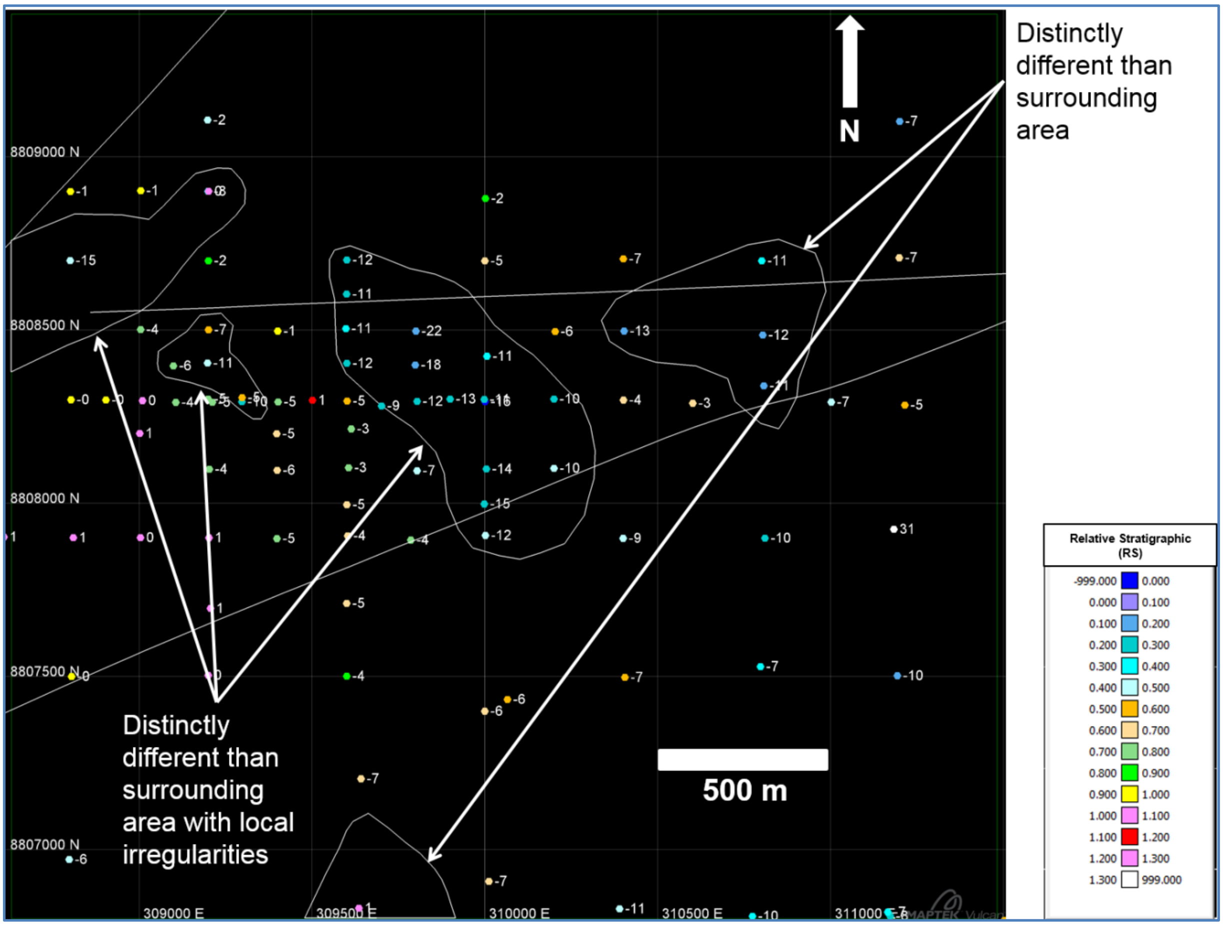


Figure provided by Ivanhoe, 2014, 1% TCu cut-off over 3 m used to select the mineralized zone (SMZ10). Scale bar represents metres.

Figure 7.30 shows an area delineated using 200 m spaced drillholes. Plotted is the distance between the base of the KPS and the centroid of the SMZ10, the mineralized intercept at a 1% ,TCu cut-off over 3 m used to select the mineralized 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-[(KPS_z – SMZ_z)/KPS_z – $ROAN_z$)]}. Again, discontinuities are present at the edges of mosaic pieces.

Figure 7.30 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 the eH-pH conditions at the time of deposition of the mineralization and /or pre-mineralization sulphide concentration in the diamictite.

7.4 Comments on Section 7

The Amec Foster Wheeler QPs note the following:

- The understanding of the deposit setting, lithologies, and geological, structural, and alteration controls on mineralisation is sufficient to support estimation of Mineral Resources and Mineral Reserves.
- Mineralisation at Kamoa has been defined over an irregularly-shaped area of 20 km x 15 km. The mineralisation is typically stratiform, and vertically zoned. The dip of the mineralised body ranges from 0° to 10° near-surface above the Kamoa dome, to 15° to 20° on the flanks of the dome. The stratigraphic position of the mineralised intercepts vary on a local basis.

- Geological controls on the mineralisation, the mineralisation style, mineralisation setting, and the mineralogy are sufficiently well understood to support declaration of Mineral Resources and Mineral Reserves.
- The occurrence of copper mineralization 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 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 postdiagenetic 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 hematite.

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 and Mufulira deposits.

Key features of the Kamoa deposit include:

- Laterally continuous, has been drill tested over an area of 20 km x 15 km.
- Associated with a 35 km-long regional structural corridor bounded by the West Scarp 1 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; association with cobalt, silver (but thus far not in economically significant quantities).
- 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.
- Occurrence of very fine-grained, bedded, disseminated copper sulphides in the intermediate sandy siltstone unit (Ki1.1.1.2) within the basal diamictite 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 Kamoa.

The Kamoa deposit is currently unique within the DRC. Kamoa is located west of the External Fold and Thrust Belt that hosts the operating mines of the Katangan sector of the Copperbelt, in that it is a large deposit that is relatively undeformed in contrast to the "écailles"-type deposits of the Fold and Thust Belt.

The Amec Foster Wheeler QPs consider the deposit model developed by Ivanhoe for the Project is appropriate to the style of mineralisation that has been identified.

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 Kamoa region, 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 prospect, situated south of the Makalu Dome. 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 (outcrop) in this area. This contact forms 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 this information has been used for locating holes for the 2015 drilling in the area.

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 Kamoa 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, DKMC_DD322, DKMC_DD325, and DKMC_DD330. The data collected included natural gamma, density, sonic, magnetic susceptibility, three component magnetics, resistivity, conductivity, induced polarization 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 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.

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.

A 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. Sulfur isotope studies indicate that most of the sulfur in the copper sulfides was derived from early diagenetic iron sulfide, and
5. Fluid inclusion analyses indicate that the mineralisation-forming fluid was saline, ~23 to 26 wt% NaCl wt equivalent, and had homogenization temperatures (Th) ranging from 210° to 240°C.

In addition, a research PhD, being conducted through the University of Toronto, is currently in progress on the depositional setting of the Kamoa deposit.

9.6 Geotechnical and Hydrological Studies

Geotechnical studies completed in support of the Kamoa 2016 PFS are discussed in Section 16.

9.7 Metallurgical Studies

Metallurgical studies and testwork completed on the Project are summarised in Section 13.

9.8 Exploration Potential

The Kamoa Project area is underlain mainly by subcropping Grand Conglomerat diamictite, at the base of which occurs at the Kamoa target, and thus the entire area underlain by diamictite can be considered prospective for discovery of extensions to the Kamoa 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 of these occur at the Kakula and Kakula Northeast Domes, which are situated along strike to the southwest, and are analogues of the Kamoa and Makalu Domes where the Kamoa mineralisation was initially discovered.

Additional drilling has been performed in the Kakula Dome Area. The 5 November 2015 database contains 39 core holes (11,466.5 m) that have been completed at Kakula. A total of 21 of these drillholes were completed in 2014.

Targets for further exploration (Kamoa-Makau and Kakula Exploration Area) are discussed in Section 14.17.

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

9.9 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 deposit.
- 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 resource estimation was closed on 5 May 2014. 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. Core holes have been used for geological modelling, and those occurring within the mining lease and in areas of mineralization (drillholes on the Kamoa and Makalu domes are excluded) have been used for resource estimation.

As at 5 November 2015, there were 1,188 core holes drilled within the broader Project area (Table 10.1). The 2014 Mineral Resource estimate used 720 drillhole intercepts. Included in the 720 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 468 holes not included in the May 2014 estimate were excluded because they were either abandoned, unmineralized holes in the dome areas, unsampled metallurgical, civil geotechnical or hydrological drillholes, or have been drilled after the closure of the database for Mineral Resource Estimation (5 May 2014), refer to Table 10.2. Subsequent to the Mineral Resource estimate, 58 drillholes have intersected the mineralised zone inside of the modelled area. Figure 10.1 shows a comparison of the TCU grades in the 720 drillholes used in the 2014 Mineral Resource estimation to the holes used in the previous Mineral Resource estimation.

Figure 10.2 shows drillholes occurring inside the Mineral Resource definition area. Core holes typically commence collecting cores at PQ size (85 mm), reducing to HQ size (63.5 mm), and where required by ground conditions, further reducing to NQ size (47.6 mm).

Table 10.1 Drilling Statistics per Category for Core Holes

Drill_Purpose	Count (Active)	Metres (m)
Resource	761	194369.8
Condemnation	51	1177.8
Exploration	48	15815.5
Metallurgy	216	89136.3
Geotechnical	22	6101.6
Variability	43	20454.6
Civil_Geotechnical	39	890.8
Cover_Drilling	3	471.5
Permeability	5	30.0
–	1188	328447.9

Note: In this table, individual wedges are considered to be separate drillholes. Meterage summary for wedge holes includes both the wedge and the pilot hole portion.

Table 10.2 Drilling statistics for drillholes not used in Mineral Resource Definition Holes

Drill_Purpose	Count	Metres
Resource	224	43,939.7
Exploration	42	11,664.9
Metallurgy	72	42,353.2
Condemnation	51	1,177.8
Variability	24	11,178.1
Civil_Geotechnical	39	890.8
Geotechnical	8	656.6
Cover_Drilling	3	471.5
Permeability	5	30.0
Total	468	112,362.5

Note: In this table, individual wedges are considered separate drillholes. 'Exploration' holes refer to those holes outside of the modelled mineral resource area.

Figure 10.1 2013 and 2014 Drill Data with a Comparison of the 2013 and 2014 Grade Distributions

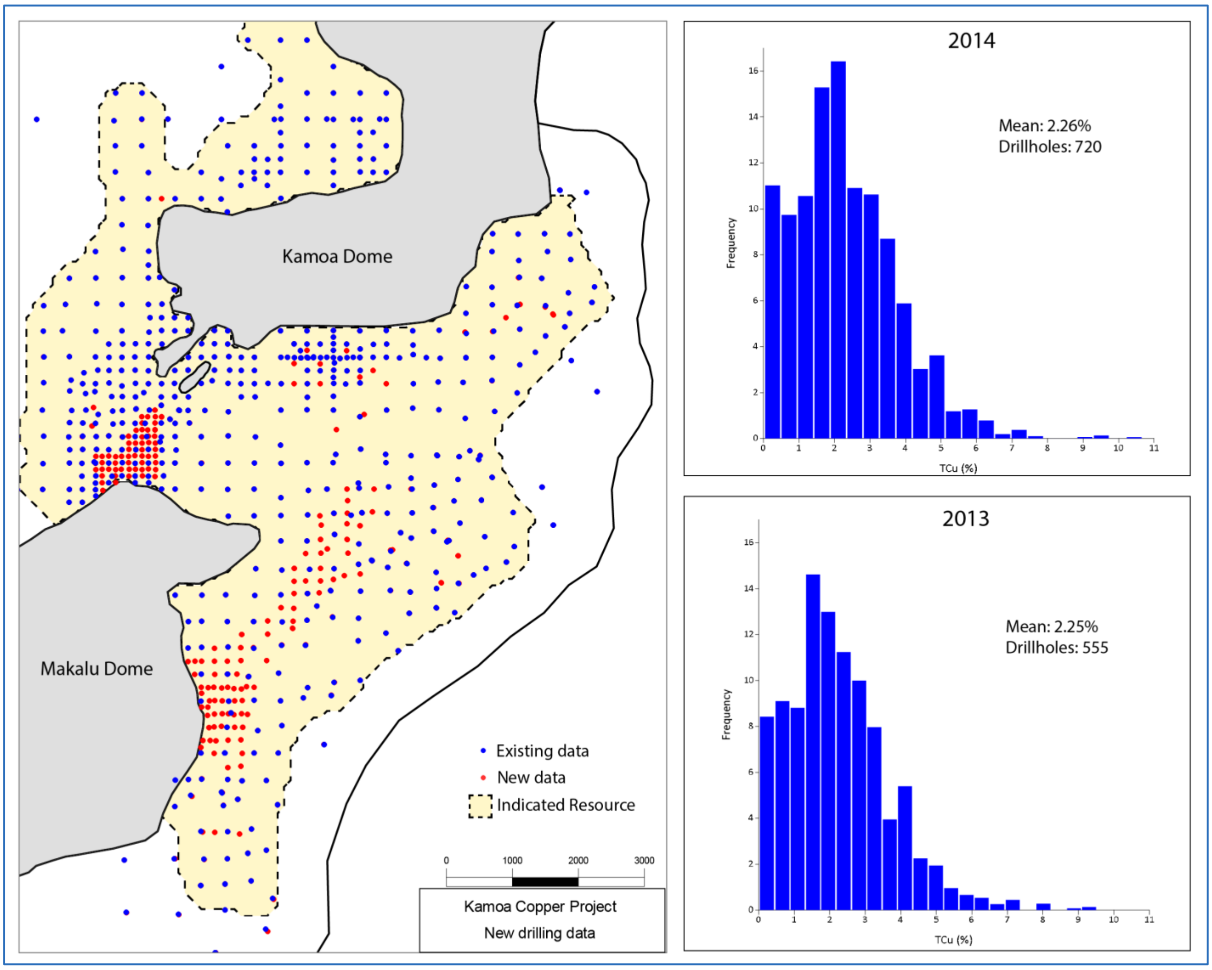


Figure provided by Ivanhoe, 2014; TCu distributions from declustered data using 400 x 400 m cell size. Scale bar in plan map represents metres.

Figure 10.2 Mineral Resource Definition Drilling at Kamoa

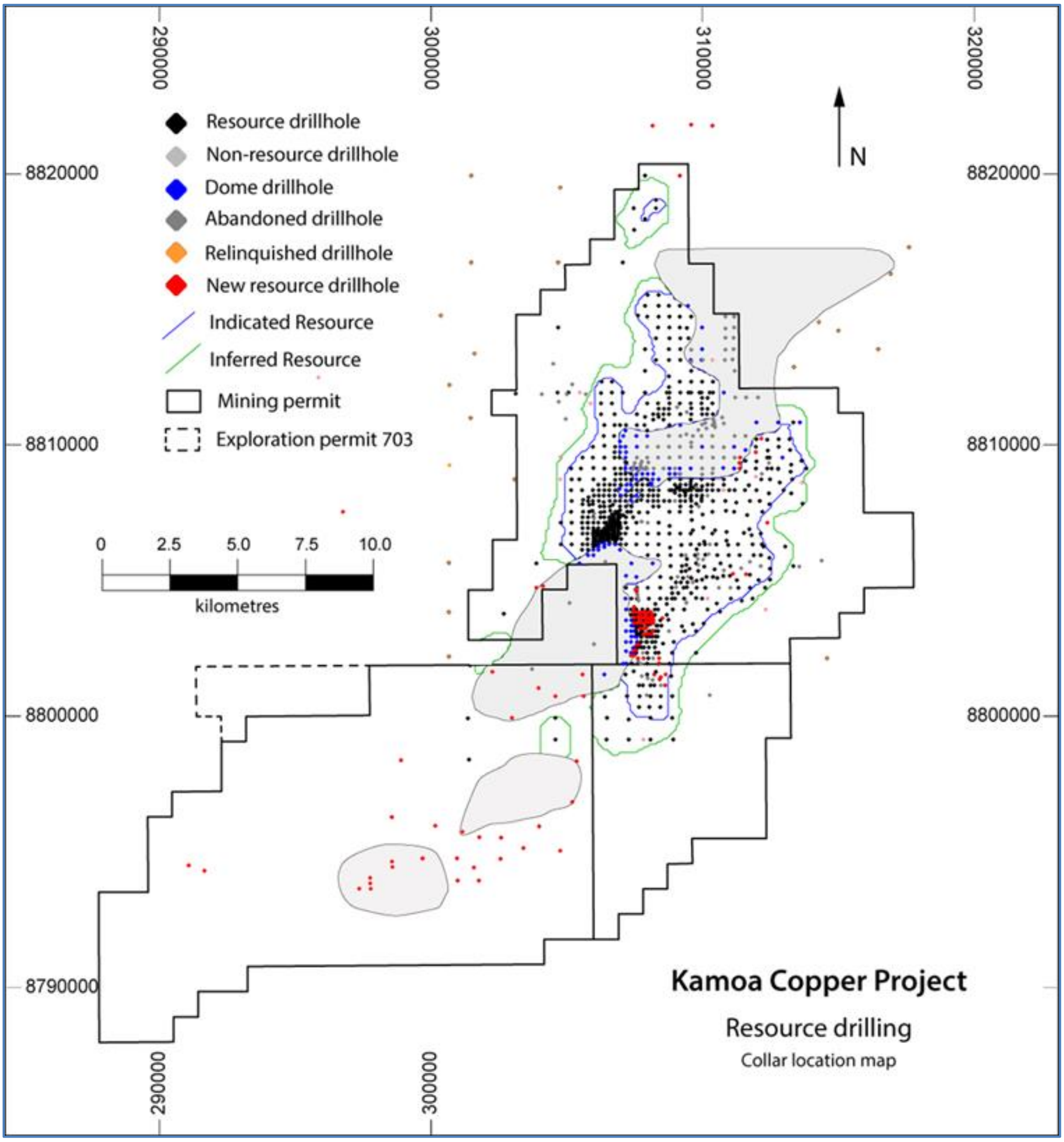


Figure provided by Ivanhoe, 2016; green line delineates mineral resource model; blue green line delineates outer limit of Mineral Resources. Scale bar represents metres.

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.

Drill core, RC, and aircore chips are logged by a geologist, using paper forms, which capture lithological, weathering, alteration, mineralisation, structural and geotechnical information. Logged data are then entered into Excel spreadsheets using single data entry methods. A hand-held Niton XRF instrument has been used from 2007 onwards on the prepared sample pulps 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. Core holes were logged at the core shed located in Kolwezi until 2009; following this all logging was moved to the Kamoa drill camp.

All drill core is photographed both dry and wet prior to sampling. All core is subject to magnetic susceptibility measurements.

During 2012, a new logging scheme was implemented to streamline the logging process, 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.

A revised “Major Structure” logging scheme was initiated based on recommendations by SRK Consultants. These data complement data recorded on the logging form described above and are captured electronically. Historic data from oriented core logging has been integrated with these data.

One sample from each core run is now subjected to magnetic susceptibility, specific gravity (SG), spectral gamma and point load testing.

10.3 Core Handling

Refer to the 2013 Technical Report for detailed description of the core handling. No changes to the handling of core has been implemented since 2013.

10.4 Recovery

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

10.5 Collar Surveys

All drill sites were surveyed using a hand-held GPS that is typically accurate to within about 7 m.

Site protocal states that all completed holes will be surveyed by an independent professional surveyor, SD Geomatique, using a differential GPS which is accurate to within 20 mm. As of 5 November 2015, a total of 29 holes remain to be surveyed (all of which post-date the resource estimate). It is expected that SD Geomatique will complete the collar surveys in due course.

10.6 Downhole Surveys

Core hole orientations ranged from azimuths of 0° to 360°, with downhole inclinations that ranged from -24.5° to vertical. Most holes were vertical or subvertical, with 1,204 of the core holes having collar inclinations that ranged from -50° to vertical.

Core hole depths ranged from a minimum of 12 m to a maximum of 1,706 m, averaging about 270 m.

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

A total of 620 of the 720 core holes used in resource modelling (see Section 14) were vertical holes (inclination less than -80°) ranging in total depth from 39 m to 1,706 m, and have end-of-hole deviations averaging 18 m in easting and 7 m in northing. The remaining 100 holes used in resource modelling were inclined with inclinations ranging from -80° to -24.5°. The average deviation for the inclined holes was 0.9° per 100 m.

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

10.6.1 Oriented Drill Core

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

As of 5 November 2015, the database contained 22 geotechnical drillholes (6,101.6 m) and 39 civil geotechnical drillholes (890.8 m).

Collar locations of geotechnical drillholes completed as at 5 November 2015 are shown in Figure 10.3.

Figure 10.3 Geotechnical Drill Collar Plan

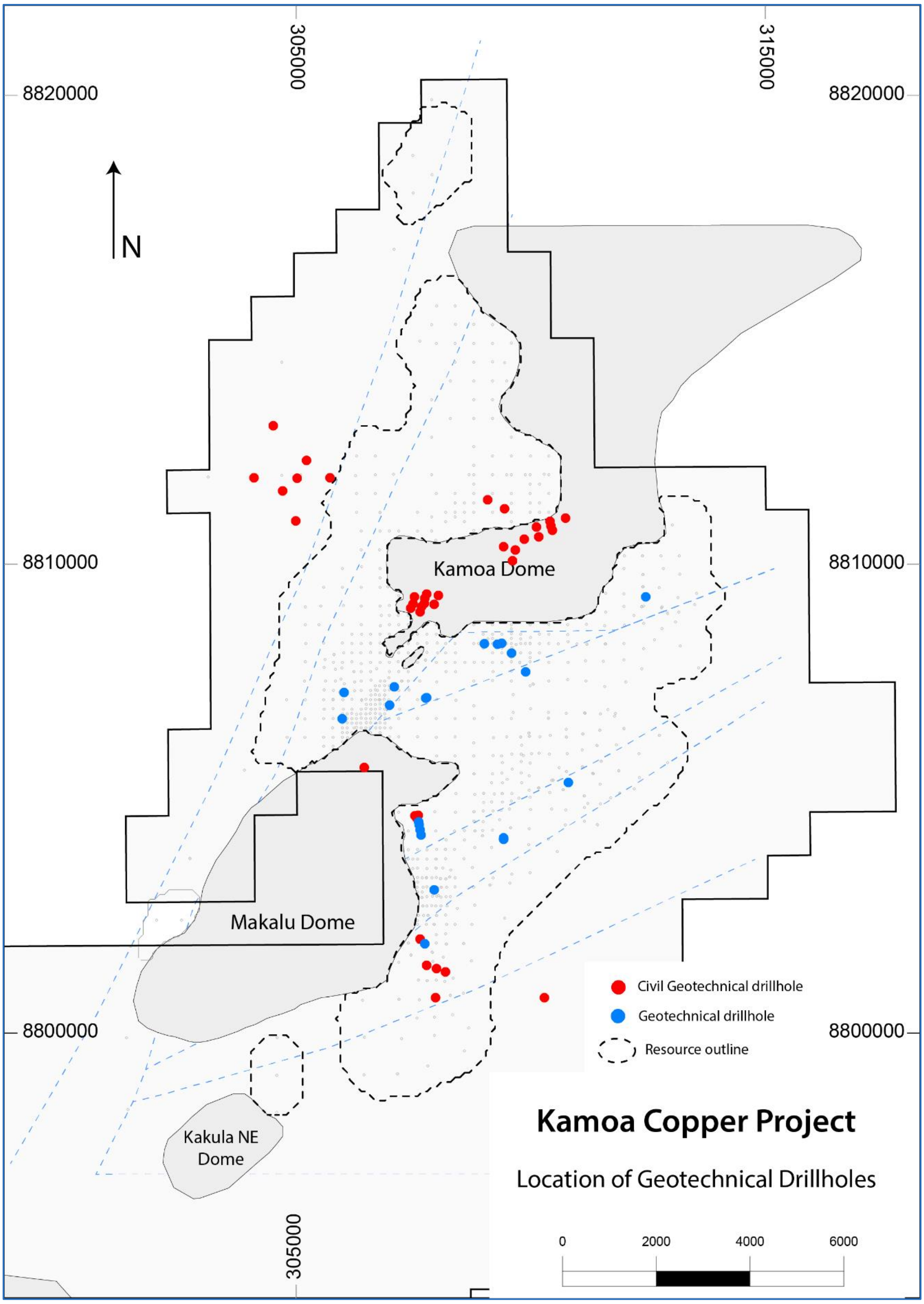


Figure provided by Ivanhoe, 2016. Scale bar represents meters. Dashed blue lines represent faults.

10.7 Hydrogeological Drilling

The location of hydrogeological drilloles are shown in Figure 10.4. Since the 2013 Technical Report Ivanhoe completed 5 permeability drillholes (30 m) in 2014.

Figure 10.4 Hydrogeological Drillhole Location Map

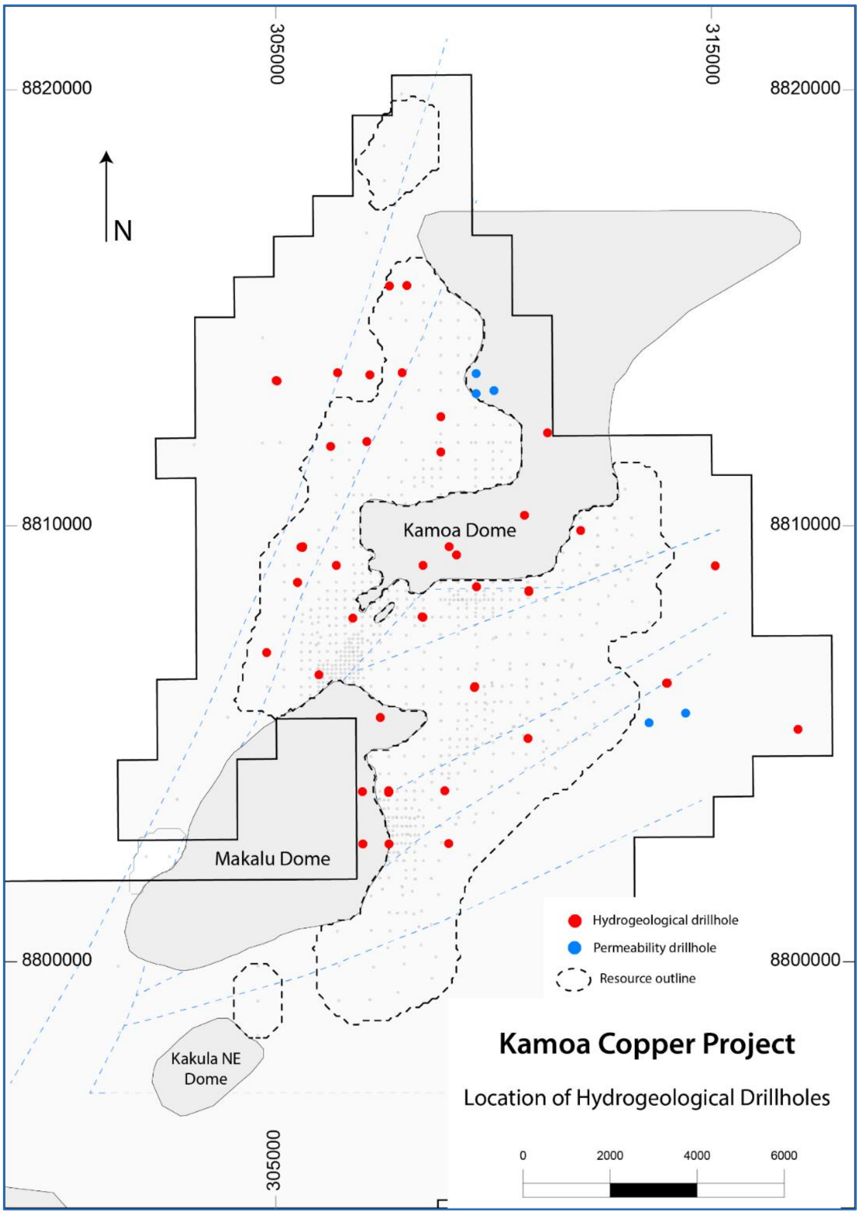


Figure provided by Ivanhoe, 2016. Dashed blue lines represent faults.

10.8 Metallurgical Drilling

The Project database contained 23 metallurgical holes (total depth 4,599 m) as of 11 March 2013. An additional 52 metallurgical holes were completed between 12 March 2013 and 14 August 2013 (total depth 12,728.8 m). As of 11 November 2015, there were 216 drillholes (89,136.3 m), this includes wedge drillholes defined as Metallurgy. The length of the wedge drillholes includes the pilot hole. Drill hole collar locations are shown in Figure 10.5.

Figure 10.5 Metallurgical Drillhole Location Map

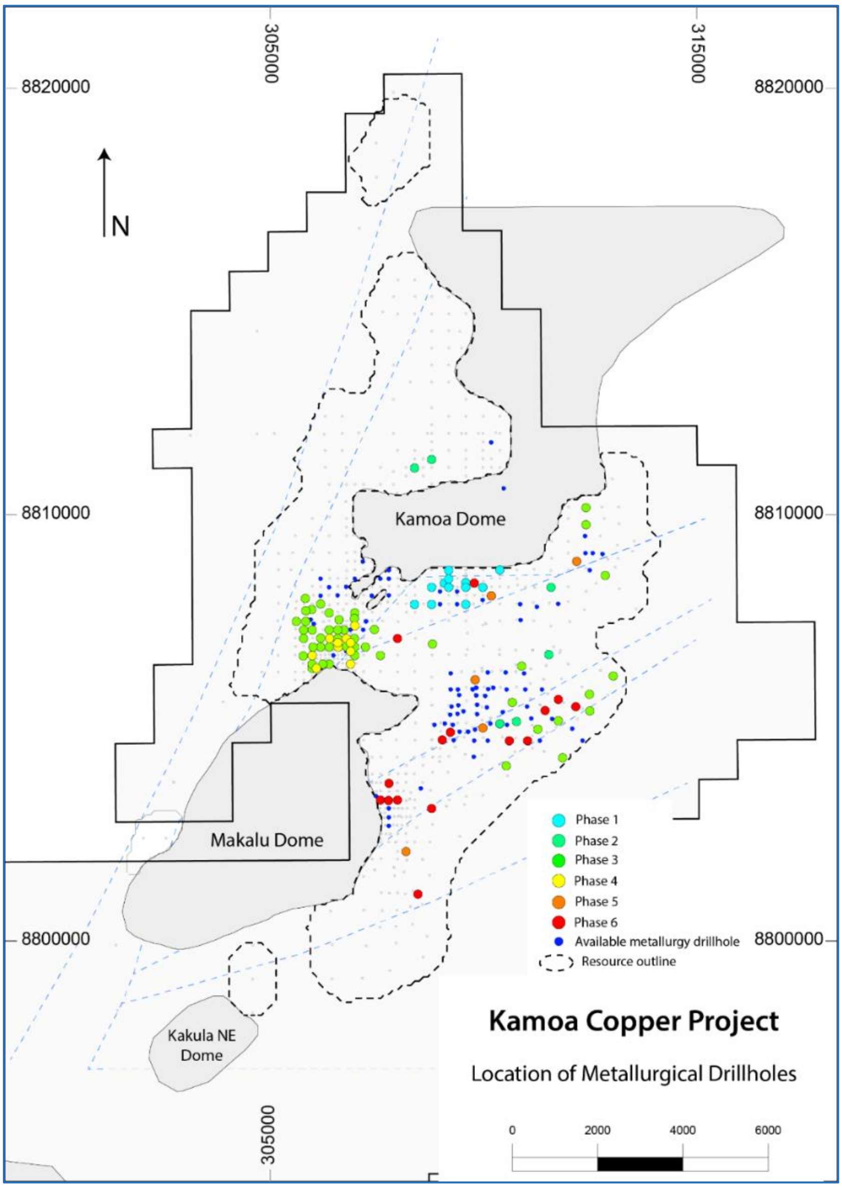


Figure by Ivanhoe, 2016. Scale bar represents metres.Dashed blue lines represent faults.

10.8.1 Variability Drilling

Since the 2013 model, significant emphasis has been placed on metallurgical and variability drilling, primarily in areas that are likely to be mined in the first 15 years of production.
Two metallurgical phases of drilling were completed to select a sample for the first four years of mining (sample 6A) and for areas planned to be mined in years 5 to 15 (sample 6B). Up to eight wedges were drilled from drillholes selected for these sampling programmes. Individual wedges were assigned for different purposes as shown in Figure 10.6, and thus only a single wedge, or occasionally a second wedge, were sampled for mineral resource estimation purposes.

A single wedge was drilled from drillholes in the Phase 6 testwork program. These drillholes were selected across the deposit. Analysis and testwork is ongoing Analysis of these samples and metallurgical testwork is ongoing. The distribution of Phase 6 holes within the Project area is shown in Figure 10.5 above.

Figure 10.6 Assignment of Individual Wedges used in the Metallurgical Testwork Programmes

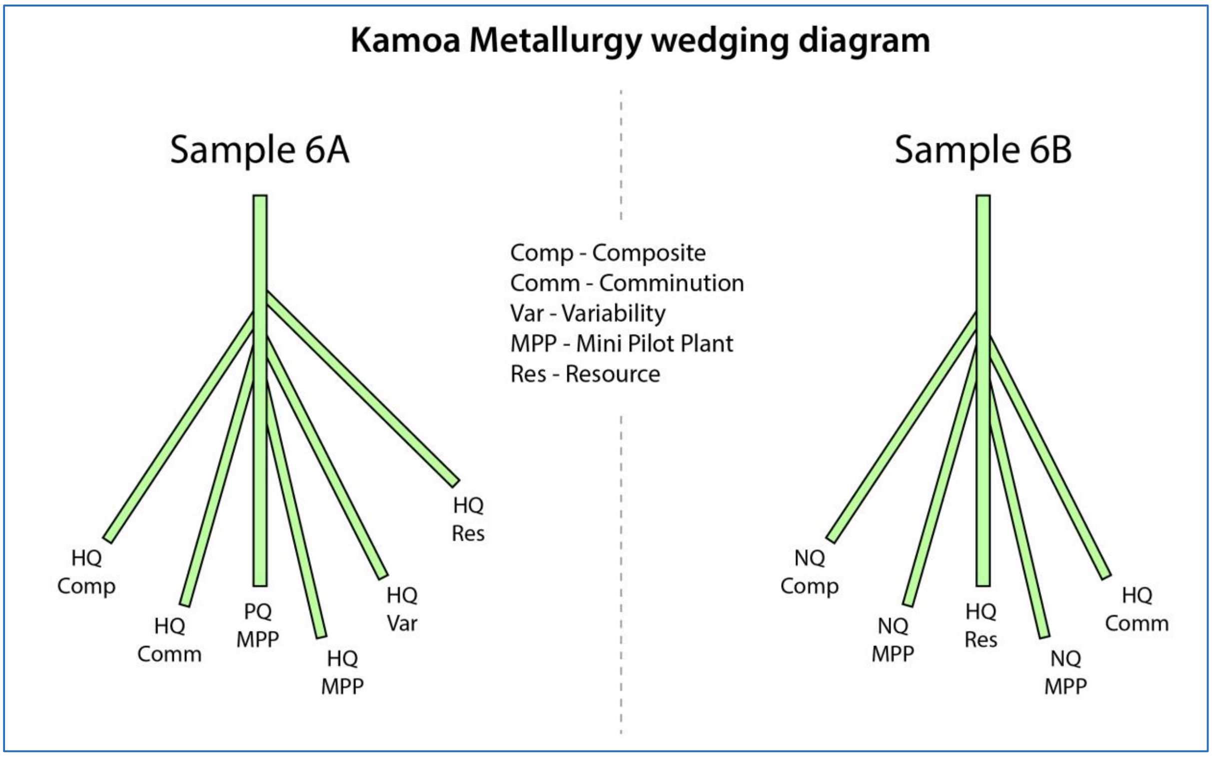


Figure provided by Ivanhoe, 2016.

10.9 Sample Length/True Thickness

Drill intersections, due to the orientation of the drillholes, are typically greater than the true thickness of the mineralisation; however, for the majority of mineralised intercepts a reduction of less than 10% in the intercept length is required to correct to true thickness (Figure 10.7). Intercepts in vertical drillholes, due to the shallow to flat dip of the mineralisation, will approximate true thickness; the 100 holes that were inclined at 25° to 80° will have drilled intersections that are greater than the true thickness of mineralisation. In some cases the "angle" holes have been purposely drilled sub-perpendicular to the mineralisation, and for these holes the intercepted and true thickness will be similar.

Figure 10.7 Drillhole Intercept Length versus Estimated True Thickness

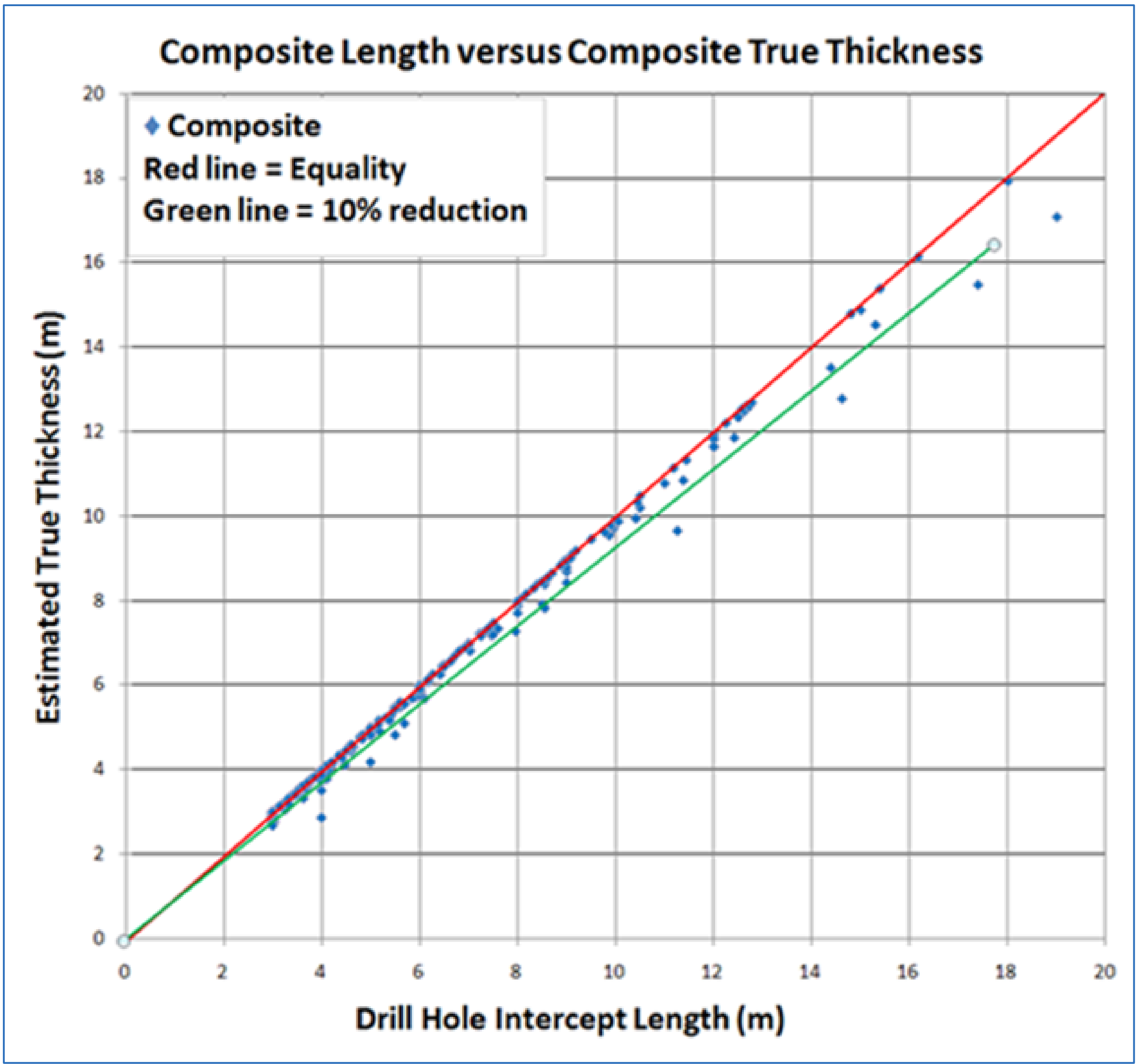


Figure by Amec Foster Wheeler, 2013. Based on mineralised intercepts used for the September 2011 Mineral Resource Estimate.

10.10 Drilling Since the Mineral Resource Database Close-off Date

The database supplied to Amec Foster Wheeler was closed as of 5 November 2015. This database contains 137 drillholes dated 2014 and 2015 that were not used in the 2014 resource estimate. Of these, 42 are dated between 16 January 2014 and 29 April 2014. Assays are now available for 15 of these drillholes. A total of 14 of these drillholes are defined as Resource and one is defined as Exploration. Of the remaining 27 drillholes, 24 are defined as Metallurgy and three are defined as Resource, but were abandoned and not sampled. There are 95 drillholes completed between 6 May 2014 and 16 October 2015. Assays are now available for 58 of these drillholes. A total of 39 of these holes are defined as Exploration and 19 are defined as Resource. The 37 holes without assays are defined as either Cover drilling (three), Exploration (14), Geotechnical, (four), Metallurgy (one), Permeability (five) and Resource (10).

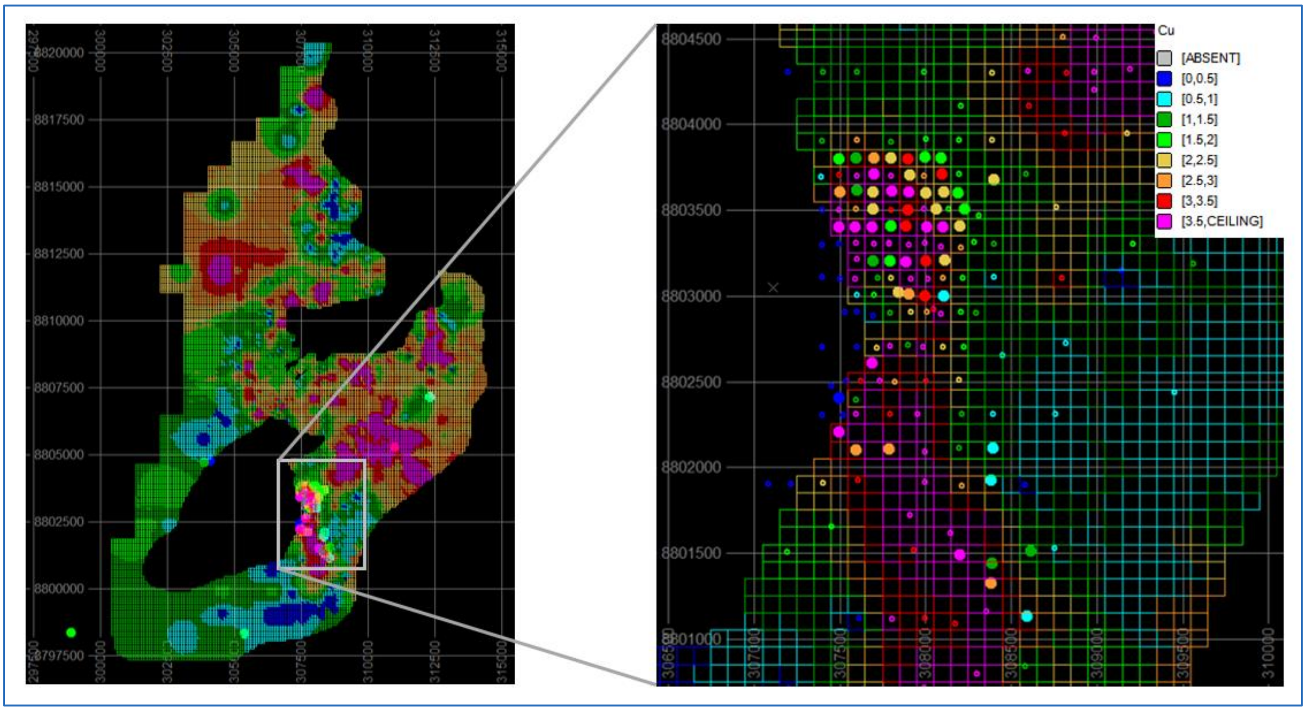
Of the 95 drillholes completed after the 2014 resource estimation, 50 were drilled at Kansoko Sud, 25 were drilled at Makalu, and seven at Kansoko Nord. The remaining 13 holes were drilled at other prospects.

Although the newer drilling within the resource model will change the grades locally, overall the new drilling should have a minimal effect on the average grade of the model (Figure 10.8). Figure 10.9 shows the location of the drillholes completed since the resource model.

Table 10.3 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.

Subsuent to 5 November 2015, Ivanhoe received assays for drillholes DKMC_DD996 and DKMC_DD997 (totalling 752.15 m), drilled at Kakula. Additional information describing these two drillholes is presented in Section 14.17.2.

Figure 10.8 Plan View Showing Drillholes with Assay Results Completed Since Construction of the 2014 Mineral Resource Model



2014 Resource model is in the background; both blocks and drillholes have been color-coded according to the legend; larger circles represent holes drilled since May 2014; smaller circles in the inset represent previously drilled holes. Grid on left is at 5000 m spacing, grid in right image is at 500 m spacing.

Figure 10.9 Plan View Showing Drillholes Completed Since Construction of the 2014 Mineral Resource Model

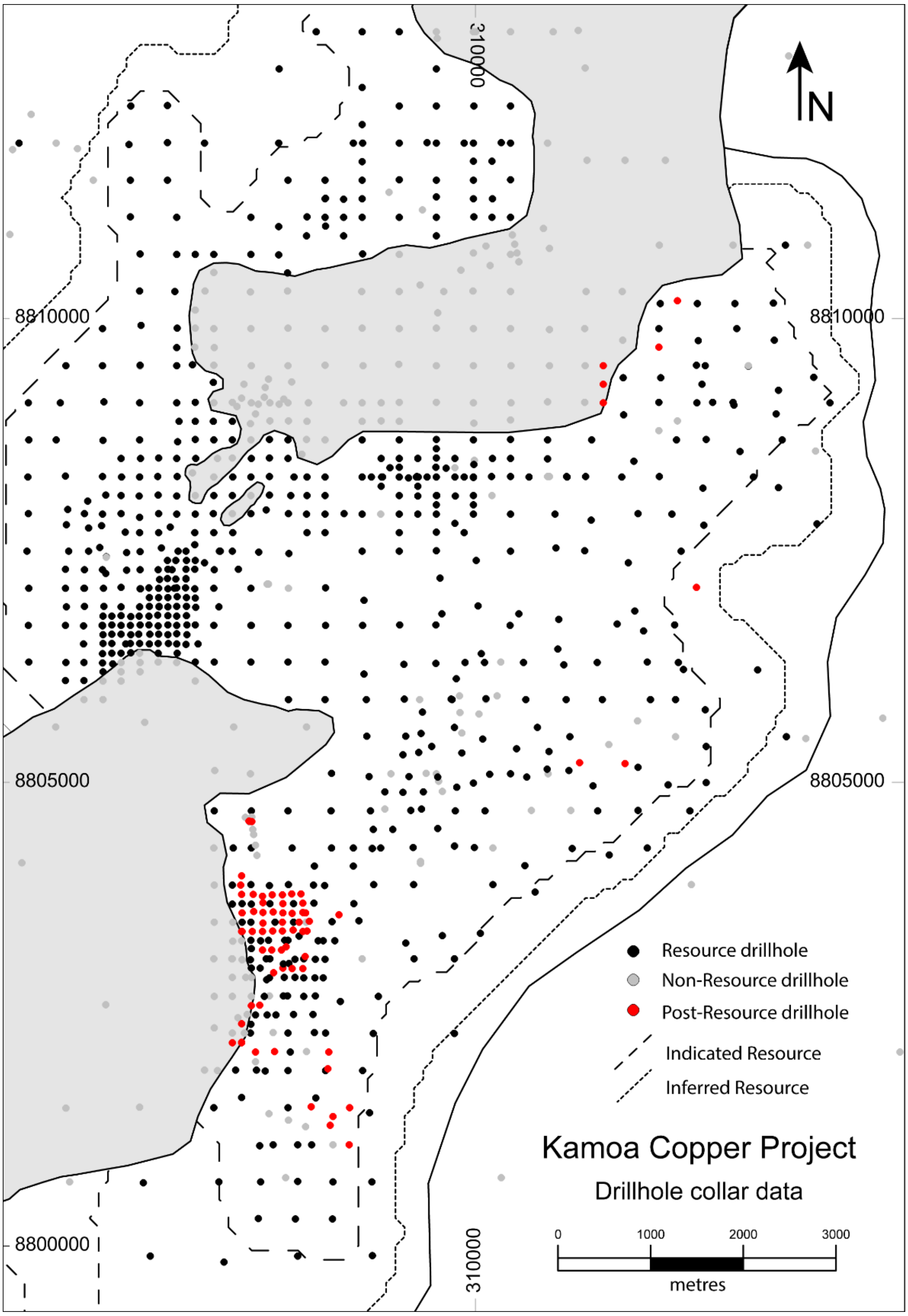


Figure by Ivanhoe, 2016.

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Table 10.3 Example Drill Intercept Table, Holes Drilled Since May 2014

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_DD903	307598.16	8802100.58	1269.056	273.7	-74.3	237.6	185	196.1	11.10	10.68	2.83
DKMC_DD915	308618.2	8801507.62	857.457	300.3	-83.4	582.5	565.5	569	3.50	3.48	1.22
DKMC_DD923	307498.83	8803797.9	1367.09	270.0	-90.0	134	104	108.5	4.50	4.50	1.85
DKMC_DD931	308229.65	8803504.79	1152.347	208.5	-88.9	354	317	328	11.00	11.00	1.56
DKMC_DD951	308191.39	8803596.69	1188.15	15.1	-89.0	321	285	290	5.00	5.00	1.90
DKMC_DD968	307800.16	8803799.93	1292.17	285.5	-81.2	213.5	181	188.43	7.43	7.34	2.41
DKMC_DD971	308095.44	8803801.35	1235.317	269.7	-80.1	276.5	243	247.5	4.50	4.43	1.87

10.11 Comments on Section 10

In the opinion of the Amec Foster Wheeler 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 as follows:

- Drill intersections, due to the orientation of the drillholes, are typically slightly greater than the true thickness of the mineralisation.
- Drillhole orientations are generally appropriate for the mineralisation style.
- Core logging meets industry standards for sediment-hosted copper exploration.
- Collar surveys were performed using industry-standard instrumentation.
- Downhole surveys provide appropriate representation of the trajectories of the core holes.
- Core recoveries are typically excellent.
- The SMZ can include both lower- and higher-grade mineralization; however, 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.
- No material factors were identified with the data collection from the drill programmes that could affect Mineral Resource estimation.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Witness Sampling

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

11.2 Sampling Methods

11.2.1 Geochemical Sampling

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

- Stream-sediment samples were collected, dried and sieved. Sub-samples were submitted for analysis;
- Soil samples were collected from the B horizon depth (30 cm to 40 cm), dried and sieved. The sieved sub-samples were submitted for analysis; and
- 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 superceded by drill data.

11.2.2 RC Sampling

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

11.2.3 Core Sampling

The core sampling procedure is as follows:

Sampling positions for un-oxidised core are marked (after the completion of the geotechnical logging) along projected orientation lines.

Pre-February 2010, determination of the sample intervals took into account lithological and alteration boundaries. The entire length of core from 4 m (or one core-tray length whichever is convenient) above the first presence of mineralisation and/or the mineralised zone is sampled on nominal whole 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 are 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 analysed by Niton. The results are used to characterise the geochemisty 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.
- 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 a standard diamond saw. The cut line (for splitting) is typically offset from the core orientation line by 2 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.
- 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 facility is comprised of three lockable buildings with 24-hour security personnel in place.

11.3 Metallurgical Sampling

The Mintek metallurgical samples were selected from available coarse reject material obtained from the core hole assay sample preparation. This material is prepared from the sawn drill core which is 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 is 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.

11.4 Specific Gravity Determinations

Specific gravity (SG) measurements were performed using a water immersion method by Ivanhoe personnel. Samples were conventionally weighed in air and then in water.

A total of 13,021 SG measurements were performed on samples taken from drill core. Of these measurements, there are 12,987 samples with SG values between 1.5 and 4.0.

11.5 Analytical and Test Laboratories

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

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

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

ALS Chemex of Vancouver, British Columbia, acted as the check laboratory for drill core samples from part of the 2009 program and for 2010 through 2013 drilling. ALS Chemex is ISO:9001:2008 registered and ISO:17025-accredited.

11.6 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 a TM Terminator Jaw crusher, Labtech Essa LM-2 pulveriser and a riffle splitter.

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

About 5% (approximately one in 20) of the crushed samples have a 2 mm screen test performed, and a further 5% at the pulverization 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.

11.7 Sample Analysis

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

11.7.1 Bureau Veritas (formerly Ultra Trace) Laboratory

Bureau Veritas acquired Ultra Trace in 2007. As the assay certificates continue to be certified by Ultra Trace, Amec Foster Wheeler refers to Ultra Trace throughout this report.

Diamond drillhole samples from 2008 to February 2009 were analysed for Cu, Zn, Co (ICP-OES), and Pb, Zn, Mo, Au, Ag, and U (ICP-MS) using a 4 gram 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 (ICPOES), and Ag, As, Au, Ba, Bi, Mo, Pb, Se, Te and U (ICP- MS) using a 4 gram subsample of the pulp using mixed acid digest (Ultra Trace method ICP102 (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. Amec Foster Wheeler recommends that samples obtained in 2008 and 2009 be submitted for ASCu analysis because very few (249 out of 6,640) samples from holes drilled prior to 2010 have ASCu assays. As of 5 November 2015, this had not been undertaken.

Samples 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 – 4-acid digestion with, ICP-OES), and As (Ultra Trace method ICP302, - 4-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).

11.8 Quality Assurance and Quality Control (QA/QC)

QA/QC samples are placed 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.

11.8.1 Blanks

Five materials, BLANK2005, BLANK2007, BLANK2008, BLANK2009, and BLANK2010 have been used in the Kamoa QA/QC. The year designations indicate the year the material for the blank was collected. Prior to using the blank material, a number of sub-samples were taken, and these were submitted for assay at Ultra Trace to confirm that the material was not Cu–Co-mineralised. In the opinion of Ivanhoe's consulting geochemist, Richard Carver, the material has low concentrations of the target elements Cu and Co (Carver, 2009a).

BLANK2005 was produced from quartz-rich material in South Africa. Bags of coarse quartz-rich material were collected from a field location in the DRC for use in 2007 and 2008 as blank material.

Material for BLANK2009 was collected in the Lualaba River, by Nzilo, about 40 km from Kolwezi. 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.

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

The BLANK2008 and BLANK2009 material was continued to be used for the 2010 drill programmes. BLANK2010 has been used in the 2010 through 2013 drill programmes. One blank per 20 samples was inserted prior to the samples being pulverised.

Blank samples are now placed after noted higher-grade mineralisation.

11.8.2 Duplicates

A preparation duplicate was created for every 20th sample by taking a second split following the crushing stage of the sample preparation. Amec Foster Wheeler also compiled Ultra Trace's QC control duplicates (same-pulp same-batch laboratory duplicates) from the laboratory reports. Duplicate samples are now placed within typical mineralisation.

11.8.3 Certified Reference Materials

Certified reference materials (CRMs) were 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 CRMs have been used, 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 Kamoa Project site. CRMs are inserted with a 5% insertion rate, and the CRM published value is matched to the expected mineralization grades. CRMs are placed within mineralisation to best match the surrounding material. CRMs have been created from matrix made from Kamoa core.

11.9 Databases

In early 2013, Ivanhoe implemeneted an acQuire data management database for storage of all relevant electronic data. Ivanhoe has completed validations to ensure the data integrity has been maintained during the data transfer, but Amec Foster Wheeler recommends a comprehensive audit be conducted.

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 cabinents in Ivanhoe's Kamoa site office. All paper records are filed by drillhole for quick location and retrieval of any information desired. In addition, sample preparation and laboratory assay protocols from the laboratories are monitored and kept on file. Digital data are regularly backed up in compliance with internal company control procedures. The backup media are securely stored off-site.

11.10 Sample Security

Sample security includes a chain-of-custody procedure that consists of filling out sample submittal forms that are sent to the laboratory with sample shipments to make certain that all samples are received by the laboratory. All diamond-drill core samples were processed by the Kolwezi facility, or the onsite Kamoa 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 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 down-hole 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 11.6.

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 camp. Historical core has been moved from Kolwezi to the facility at the Kamoa camp. Historical RC and aircore samples remain at the Kolwezi storage facility.

11.11 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, 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, the tests performed have been particle size appropriate, such as flotation and ball mill grindability.

12 DATA VERIFICATION

12.1 Amec Foster Wheeler Verifications (2009 – 2015)

Between 2009 and 2015, Amec Foster Wheeler conducted multiple reviews of the 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 the most recent audit was conducted in December 2015 (Spencer, 2015).

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.

In 2010, Amec Foster Wheeler suggested changes to improve the mineral abundance logging in order to reduce overestimation of sulphide mineral abundances.

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 was 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 of Acuity Geoscience Ltd. with the exception of the 2014 check assays, which were reviewed by Amec Foster Wheeler.

12.2.1 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). Only 51 results from 3,809 were below this specification.

Ultra Trace tested every 20th sample pulp by wet-screening a weighed portion through a 75 µm (200 mesh) screen. Results indicate adequate performance, with a median value of 95% passing 75 µm. A total of 760 results from 4,212 samples were below the specification of 90% passing 75 µm. If results revealed poor comminution, the samples were typically submitted for regrinding. A review of the regrind results shows results of over 90% passing 75 µm were achieved.

12.2.2 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. CRM AMIS118 was introduced in October 2009, AMIS120 and AMIS050 were introduced in February 2010, AMIS163 was introduced in July 2011, and OREAS CRM 166 was introduced in December 2010.

Due to the low numbers of Geostats CRMs submitted, Amec Foster Wheeler's analysis of QA/QC results was limited to the OREAS and AMIS CRM results. The suite of CRMs used over the course of the Project has partially changed, with addition of higher-grade CRMs to better cover the grade range for copper. Not all CRMs are certified for all the elements requested for assay, and which elements a CRM was assayed for varies with the requests of the submission in which it was placed.

Outliers observed in the plots may be a consequence of mis-identifying which CRM was inserted. In March 2013, Amec Foster Wheeler supplied Ivanhoe a list of 55 CRM samples suspected of being mislabelled. A review of the October 2013 database indicated these samples had been investigated and corrected by Ivanhoe staff.

After removing outliers, Amec Foster Wheeler plotted Ultra Trace copper results against certified values. Linear fits on the plots suggest an overall low bias of 3% over the period examined.

After excluding the outliers, Amec Foster Wheeler calculated the average result for each element for each CRM for the remaining CRM samples. Compared to certified values of OREAS 45P, Ultra Trace is biased low by 16% to 20% on all evaluated elements. OREAS describes 45P as a ferruginous soil overlying a pyroxenite/gabbro contact, mixed with a barren soil. The consistent bias across all elements suggests a cause other than calibration; for example, the material may be a hygroscopic laterite that was not dry at the time of sample weigh-out. In most other cases, biases exceeding 10% occur at very low concentrations, and the absolute difference associated with the bias is quite small.

Amec Foster Wheeler reviewed 561 CRM results obtained from December 2012 to 1 October 2013 and found only two results showing large disagreement with the certified value. These are likely due to mislabelling the CRM in the database.

In the opinion of the Amec Foster Wheeler QPs, with the exception of the OREAS 45P CRM, the overall relative bias for the various CRMs is within 5%, and the assay accuracy is sufficient to support Mineral Resource estimation.

12.2.3 Check Assays (2009 – 2014)

Previous Check Assays

Check assays 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. The agreement between Ultra Trace and Genalysis for cobalt is good; thus the cobalt assays are likely to be accurate.

From January to July 2010, sample pulps previously assayed by Ultra Trace, along with inserted blind CRMs, were submitted to ALS Chemex (Vancouver, Canada) and results for copper and cobalt agree within 5%.

In 2011, 1,052 samples were submitted to ALS Chemex (Vancouver, Canada), both copper and acid soluble copper agree within 5%.

In 2012, 1,048 samples were submitted to ALS Chemex (Vancouver, Canada), both copper and acid soluble and cobalt agree within 5%.

In 2013, 1,068 samples were submitted to ALS Chemex (Vancouver, Canada), copper agrees within 5%, while acid soluble copper agrees within 10%.

In 2014, 755 sample pulps were submitted to ALS for total copper assay. The check results show agreement within 2%.

12.2.4 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. Amec Foster Wheeler evaluated the duplicate samples by calculating the Absolute Value of the Relative Difference (AVRD), equal to the absolute value of the pair difference divided by the pair mean:

$$AVRD = 2 * | A - B | /(A + B)$$

Duplicate pairs of this type have good precision if 90% of mineralised pairs (i.e. samples with grades well above the analytical detection limit and at or above the lowest probable mineralised material–waste cut-off grade for a mining operation) agree within +/-20% (pair difference divided by pair mean). Ninety percent of 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 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 QA/QC Review

AMEC conducted a high level review of the QA/QC reports supplied by Ivanhoe Mines Ltd. staff which covered the period November 2013 – July 2014. These QA/QC reports were prepared by Ivanhoe staff and reviewed by their consultant Mr. Dale Sketchley of Acuity Geoscience Ltd. Ivanhoe submits 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, banks and duplicates within the zone of assay, and between 3 and 5% for the zone of Interest. There are always at least 2 original samples before any new QAQC 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.

Blanks 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%.

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

12.2.7 Acid Soluble Copper Determinations

In 2009 African Mining Consultants selected approximately 431 samples for acid-soluble copper (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. 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.

12.3 Site Visits

Dr Harry Parker visited the Kamoa Project from 1 to 3 May 2009, from 27 to 30 April 2010, and again from 12–14 November 2012. 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 areas, and discussions of geology and mineralisation interpretations with Ivanhoe's staff.

Mr Gordon Seibel visited the Project from 9 to 10 February 2011, from 5 to 8 November 2011, from 12–14 November 2012 and again from 18–22 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 the Discovery Target.

12.3.1 Field Drill Collar Check

Field drill collar checks were completed by Amec Foster Wheeler staff in 2009, 2010, and 2011, 2012, and 2016 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.

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 aluminum 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 12.1 and Figure 12.2 show the logging facility and core storage respectively.

Figure 12.1 On-Site Core-Logging Facility

Photograph by Amec Foster Wheeler, 2011.

Figure 12.2 On-Site Core-Storage Facility

Photograph by Amec Foster Wheeler, 2011.

12.3.3 Inspection of Drill Core

The following core holes 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 core holes were examined during the 2010 visit:

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

The following core holes 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 core holes 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 core holes 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 sit visit, the following core holes were examined from the Kakula Exploration Target 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.

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 and 2012 site visits, Amec Foster Wheeler toured the Kamoa-site sample preparation facility and was satisfied with the operation. The procedures had not changed from the previous operation.

12.4 Amec Foster Wheeler's 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 Chemex.

ALS Chemex, an independent laboratory that is located in Vancouver, British Columbia, 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. ALS Chemex is ISO9000:2008 registered and ISO 17025 accredited.

The correlation between laboratories was found to be good. The ratios of Ultra Trace to ALS Chemex 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 Jan 2016, Amec Foster Wheeler selected four sample intervals from drill core boxes from the Kakula Exploration Target 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 averaged 10% lower than the original assays.

12.5 Geotechnical and Structural Logging

SRK completed three site visits to the Kamoa Copper Project during 2011 for the purposes of geotechnical and structural logging QA/QC and data quality control. Ross Greenwood, Ryan Campbell, and Desiré Tshibanda completed geotechnical logging QA/QC, training, and data quality review during 22–27 June 2011. From 5–12 August 2011, Ross Greenwood and Desiré Tshibanda completed geotechnical logging QA/QC, reviewed changes implemented to logging practices, and conducted additional data quality reviews 12–17 August 2011, Wayne Barnett, SRK completed a review of structural data collection and the structural geology model, and provided input for future data collection and interpretation. The site visits and accompanying memoranda are considered to satisfy tasks within SRK's scoping level proposal issued to Ivanhoe on 29 May 2011.

Findings from the visits have been documented in two memoranda 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's geotechnical engineers have been recommended at all stages of data collection.
- Structural geology findings: 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.

12.6 Comments on Section 12

In the opinion of the Amec Foster Wheeler 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. 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.

Acid-soluble copper is generally low; there is a need to correlate ASCu with metallurgical recovery, particularly in partially leached zones within 100 m of the surface.

There has been no review of the QA/QC data since July 2014. These data should be incorporated into future QA/QC reviews.

A compreshenive audit of the acQuire database should be conducted to ensure the data integrity was maintained during the transfer.

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 flowsheet 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 reserve grade has increased as better ore zones were identified. Such learning and transitions are not uncommon for this style of mineralisation which is geologically. The historic sample selection and testwork, defined as Phases 1 to 5, provided the requisite metallurgical understanding to support the PEA and subsequent Technical Reports ahead of the Kamoa 2016 PFS.

In preparation for the Kamoa 2016 PFS the Phase 6 samples were selected and the associated metallurgical evaluation was conducted over 2014–2015. The Phase 6 samples best represent ores to be processed according to the early years (years 1 to 15) of the Kamoa 2016 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 Kamoa 2016 PFS mine schedule.

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 and timings of these streams within the five historical testwork phases are shown in Table 13.1.

Table 13.1 Kamoa Historical Metallurgical Testwork

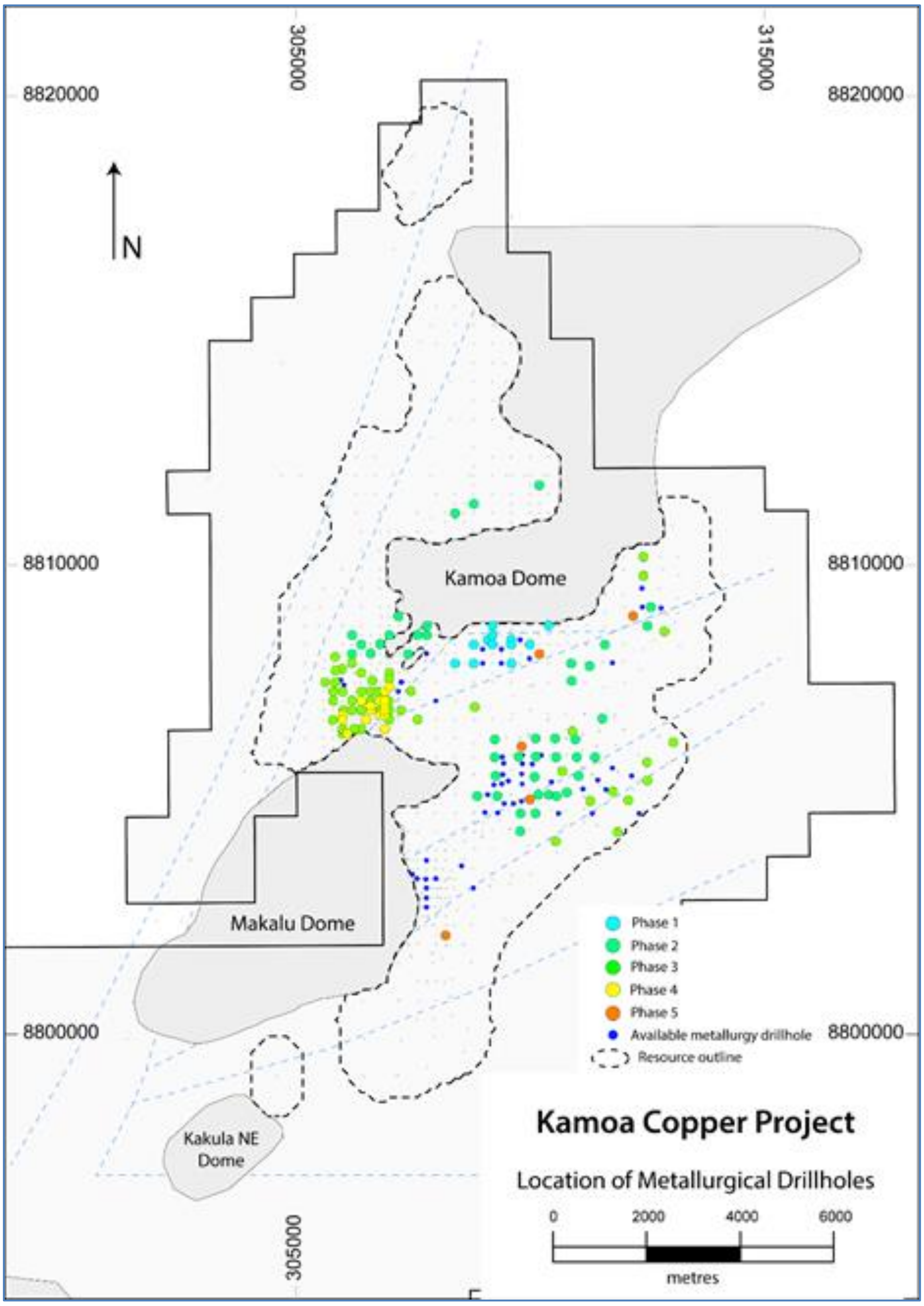
Phase	Study	Comminution	Flotation	Mineralogy	Period	Comment
1	Concept	Mintek	Mintek	SGS Johannesburg	2010–2011	Grab Samples
2	SS/PEA	Mintek	Mintek/XPS	XPS	2011–2012	Representative Composites
3	SS	–	XPS	XPS	2012–2013	Composites
4	SS	Mintek	XPS	XPS	2013	Open Pit
5	SS/PFS	Mintek	XPS/Mintek	XPS	2013–2014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

13.3 Historical Metallurgical Sample Locations

The drillhole locations that provided the historical 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 orebody, 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



13.4 Historical Comminution Testwork

The comminution test program is summarised in Table 13.2.

Table 13.2 Historical Comminution Program, Sample Numbers Tested

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

13.4.1 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 & unmineralised)	Oxide	Pyritic Siltstone (mineralised & unmineralised, hangingwall)	Sandstone (unmineralised, footwall)
1	37 to 38	–	29	–
2	22 to 31	–	21 to 22	25
4	–	44 to 58	–	–
5	17–28	–	28	30

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

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

The Phase 5 samples strongly confirmed the extreme competence of the diamictite.

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 ores exhibit different BBWI values depending on the closing screen used in the BBWI test. Where such comparative test have been done the results are shown separately.

Table 13.4 BBWI Test Results as kWh/t Value Range

Phase	Diamictites (Hypogene, Supergene & unmineralised)				Oxide		Pyritic Siltstone (mineralised & 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 hanging wall pyritic siltstone is harder than the diamictites. However, this is not the case with the Phase 5 samples. The footwall sandstone sample had similar grinding properties to the diamictites. The oxidised samples were consistently softer than the fresh samples.

In terms of sensitivity to grind size, fresh diamictite showed none, pyritic siltstone showed a reverse trend to that expected and oxide showed only a slight hardening trend.

13.4.3 Coarse Grindability (BRWI) 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 & unmineralised)	Oxide	Pyritic Siltstone (mineralised & 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 ore. 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 oversize particles in the feed. With BRWI values of 20 kWh/t the ball mill feed topsize should be limited to about 9 mm. At BRWI values of 24 kWh/t consideration should be given to lowering the screen aperture to 7 or 8 mm.

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 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}.

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 & unmineralised)	Oxide	Pyritic Siltstone (mineralised & 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 ore is significantly lower than the deeper fresh ore as expected. The average CWI for oxide ore 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 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 & unmineralised)	Oxide	Pyritic Siltstone (mineralised & 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 like 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 Ore Type

Phase	Diamictites (Hypogene, Supergene & unmineralised)	Oxide	Pyritic Siltstone (mineralised & 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 an 8 to 10 mm Ball mill feed topsize 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 One (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 benchscale 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 Copper belt 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 Flow Sheet

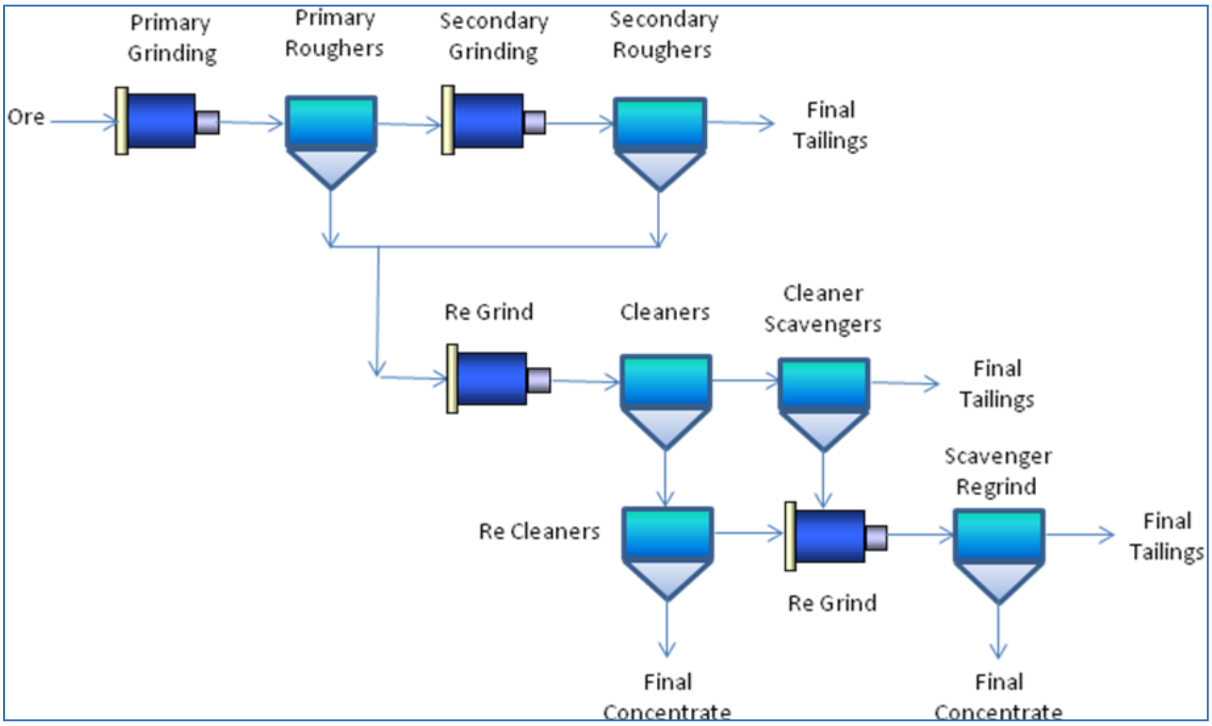


Image courtesy of Mintek 2010.

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 so the extent of 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" flow sheet milling to 80% passing 75 µm, followed by rougher flotation and two stages of concentrate cleaning. The rougher tails were then reground and subjected to a scavenger flotation stage.

Phase 2 testing showed the:

- Mineralisation tested from other zones of the Kamoa deposit responded in a similar way to the Phase 1 samples, confirming that the 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 indicted.

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 2016 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 which 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

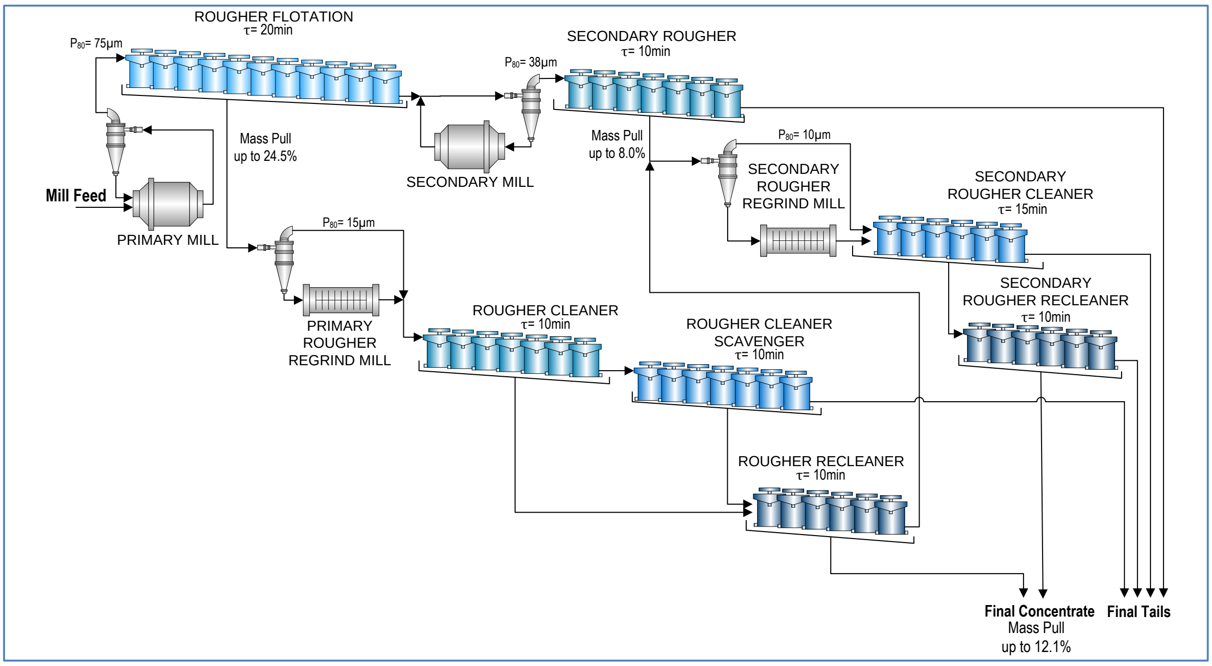


Figure by Hatch, 2013.

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

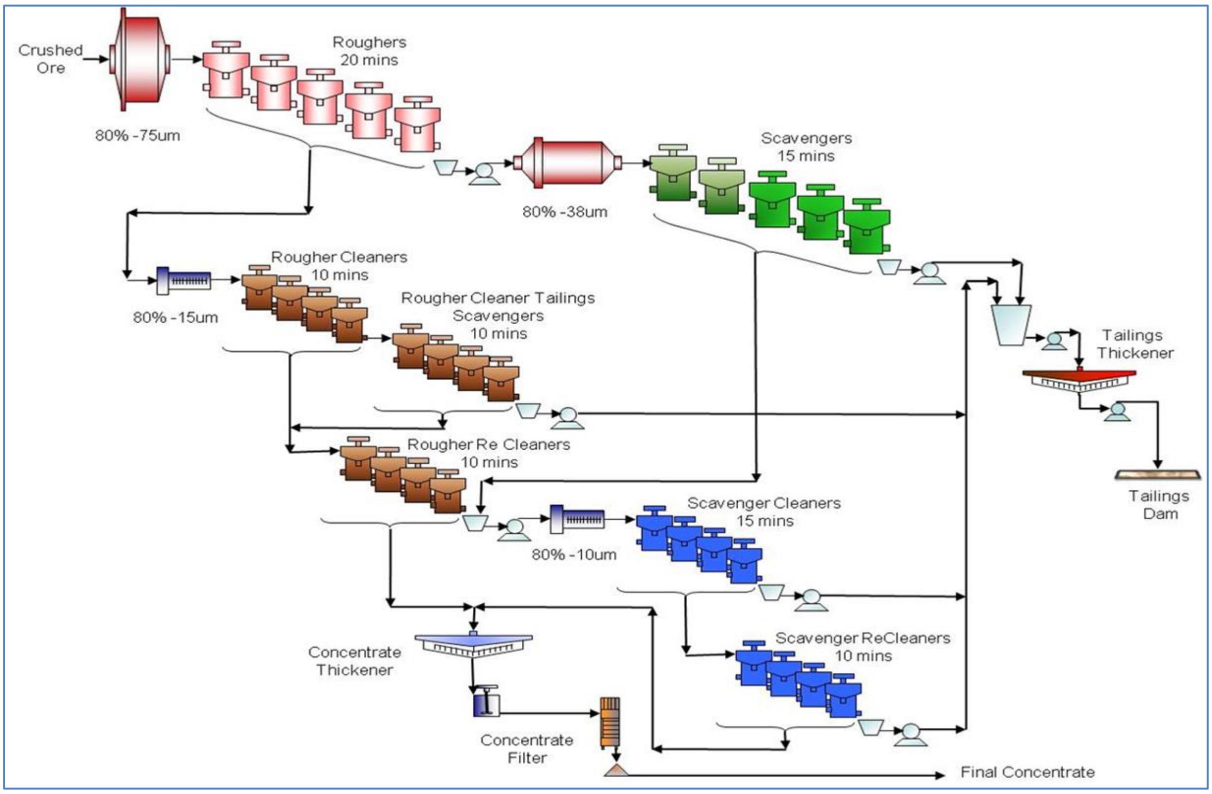


Image courtesy XPS, 2013.

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 account for sulphide sulphur.
- 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 impacted 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 2016 PFS mine schedule so these results do not influence the process results.

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 SiO2 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 primary re-grind mill from the circuit will result in low Cu grades and high SiO2 content in the final concentrate. 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 SiO2 content in the final concentrate. This confirmed that the optimum grind for re-grind circuit was P_{80} of 15 µm and 10 µm for primary and secondary re-grind mills respectively.

- Effect of alternate grind test indicated that milling finer in the secondary mill increases Cu recoveries however this is accompanied by high SiO2 entrainment. The secondary cleaner circuit optimisation will be required to reduce SiO2 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 2016 PFS Design Testwork

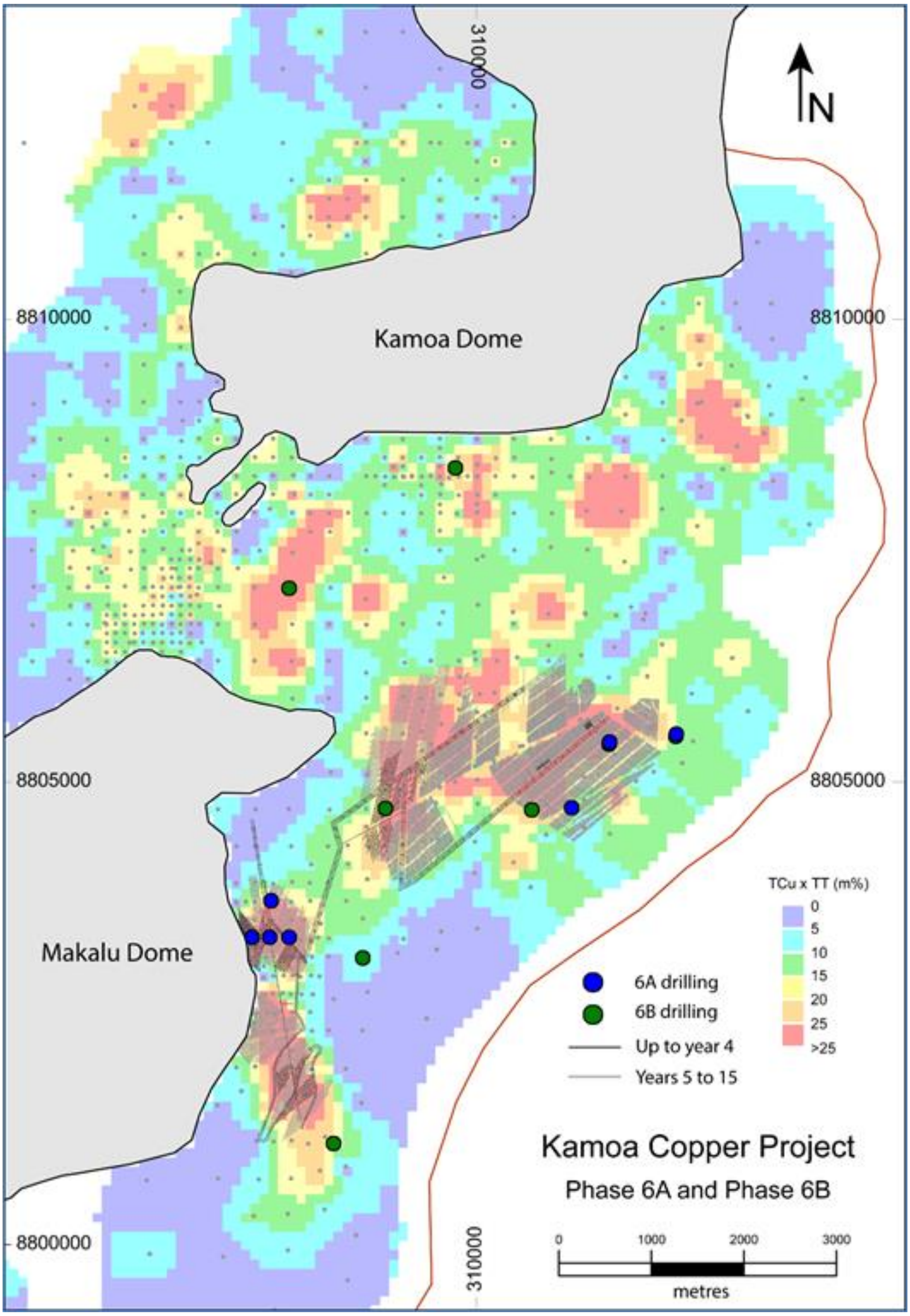
To support the PFS new samples were collected from probable mining areas. These samples were subjected to comminution testing at Mintek and flotation testing at XPS.

13.6.1 Phase 6 Comminution Testwork - Mintek

New 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 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.

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



The Phase 6 comminution results are shown in Table 13.10.

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

Ore 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 so 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) have CWI values twice that of the Phase 6 samples at 18.6 and 19.6 kWh/t respectively.

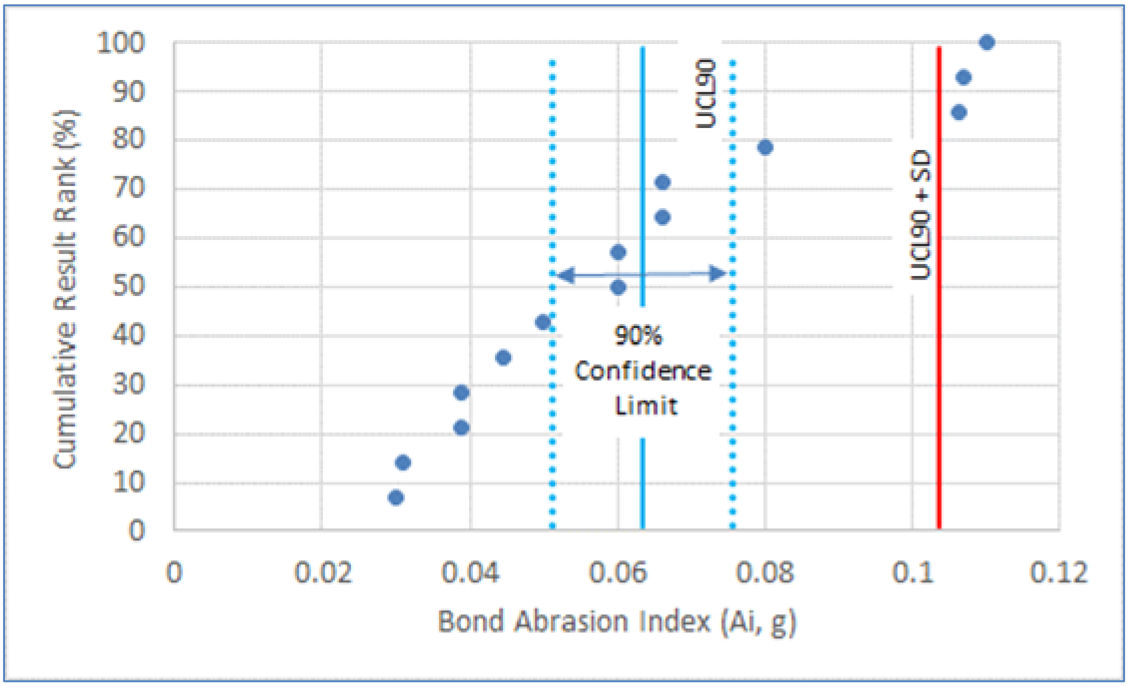
The Kamoa 2016 PFS basis of design (BOD) uses the comminution properties in Table 13.12. An appropriately high CWI value has been selected.

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

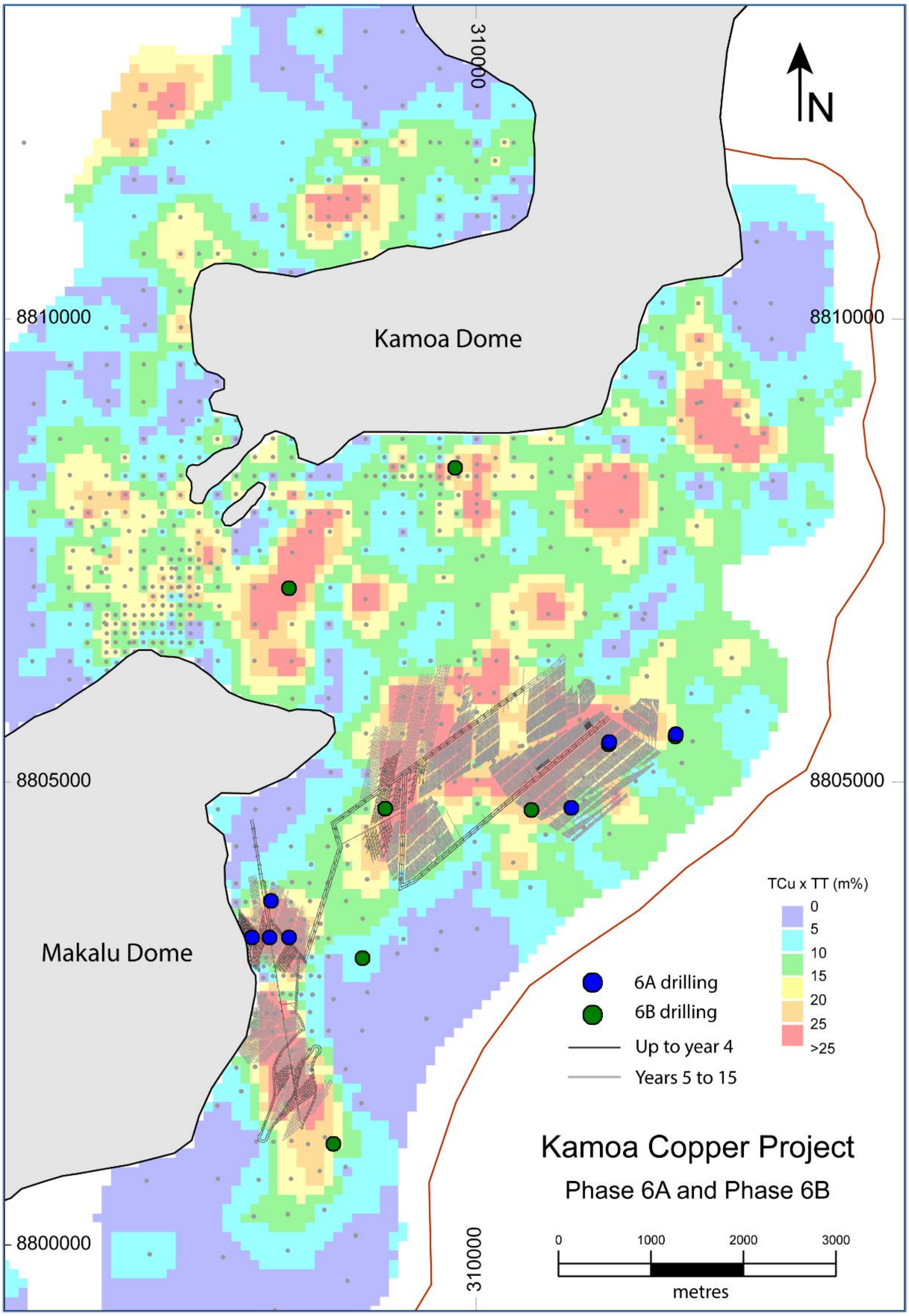
Figure 13.6 UCL90 Determination for Ai



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 so 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 required 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 13.7.

Figure 13.7 Drill Collars for Phase 6 Flotation Test Composite Samples



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 a similar answer 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 Kamoa 2016 PFS mine schedule (years 0–15).

Composites representing years 0–4 were tested under the label Phase 6A and composites representing years 5 to 15 were labelled Phase 6B. 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

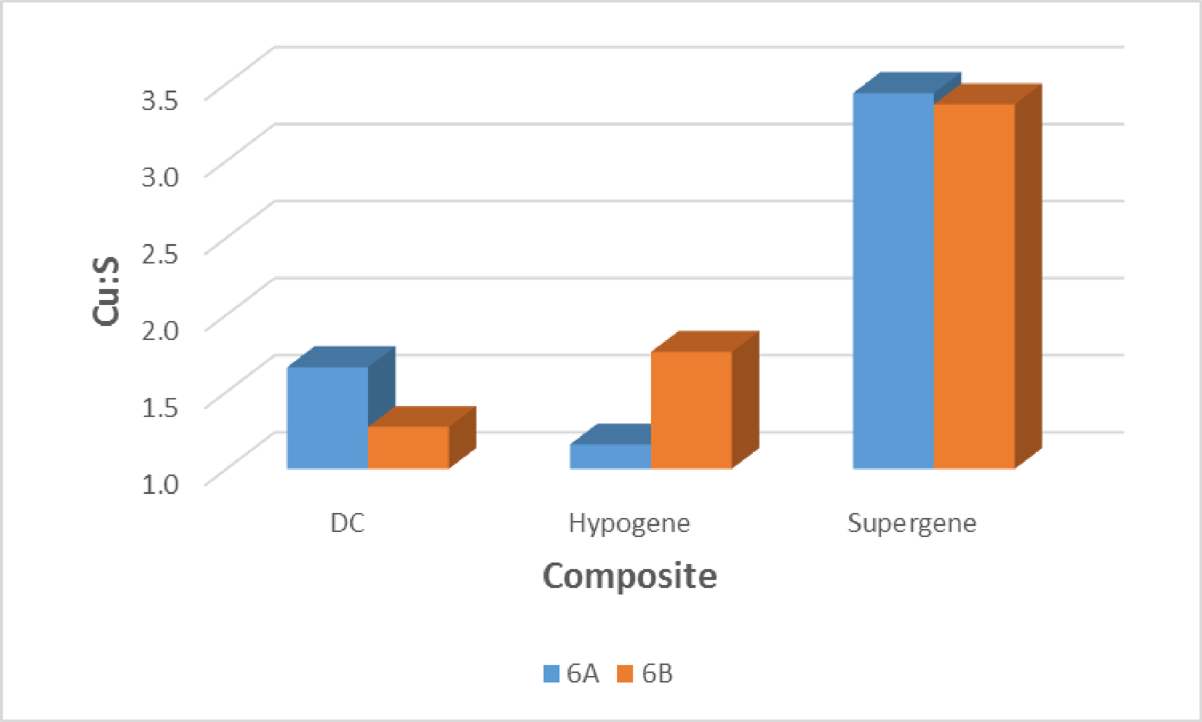


Image courtesy of Amec Foster Wheeler, 2016.

Normally hypogene ore 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 like 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

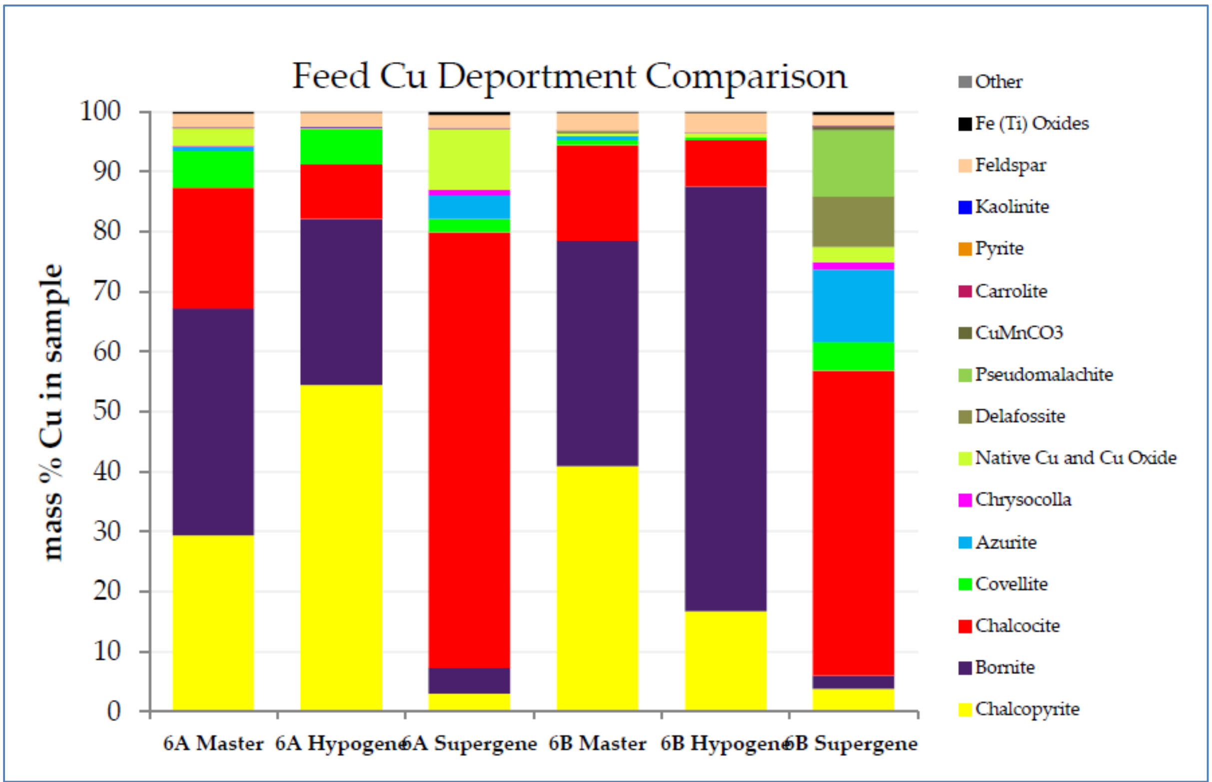


Image courtesy XPS, 2015.

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 ore. The purple band representsbornite 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 2016 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.

Figure 13.10 XPS IFS4 Flowsheet

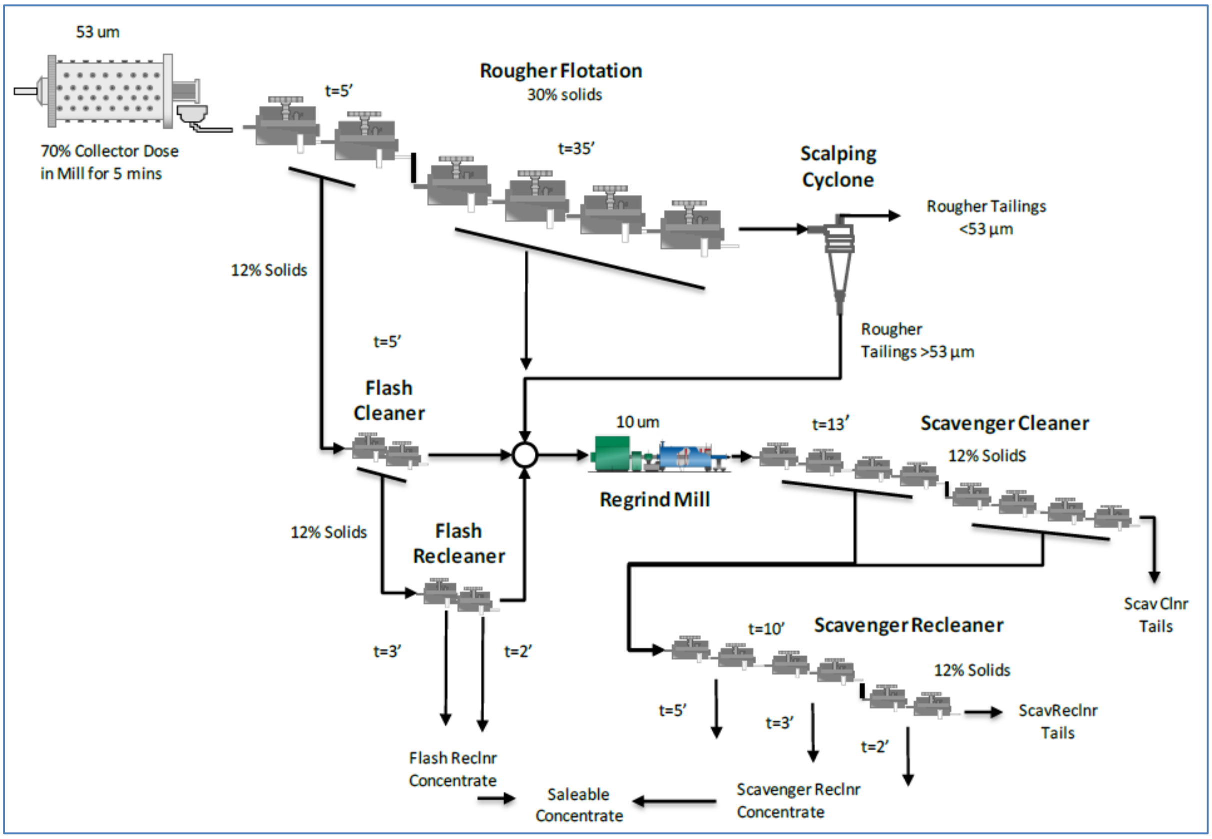


Image courtesy XPS, 2015.

Each of the six primary Phase 6 composites was tested using this flowsheet and the results are compared in Table 13.14.

Table 13.14 Flotation Results – IFS4 Circuit

		Final Concentrate					Tail	Feed
Composite		**Mass%**	**% Cu**	**Rec Cu (%)**	**%SiO2**	**%Fe**	**% Cu**	**% Cu**
6A	DC	8.53	39.0	88.3	14.6	16.3	0.48	3.76
6A	90:10 H:S	8.75	37.2	88.7	6.13	22.9	0.45	3.58
6A	Hypo	8.98	35.7	90.0	4.92	23.4	0.40	3.56
6A	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
6B	Hypo	6.29	44.5	91.9	10.6	15.4	0.26	3.05
6B	Super	5.96	46.5	69.4	15.8	10.6	1.30	3.99
15 yr 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

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

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

The 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 to 23.5 kWh/t for IFS4a.

Figure 13.11 XPS IFS4a Flowsheet – Basis of the Kamoa 2016 PFS

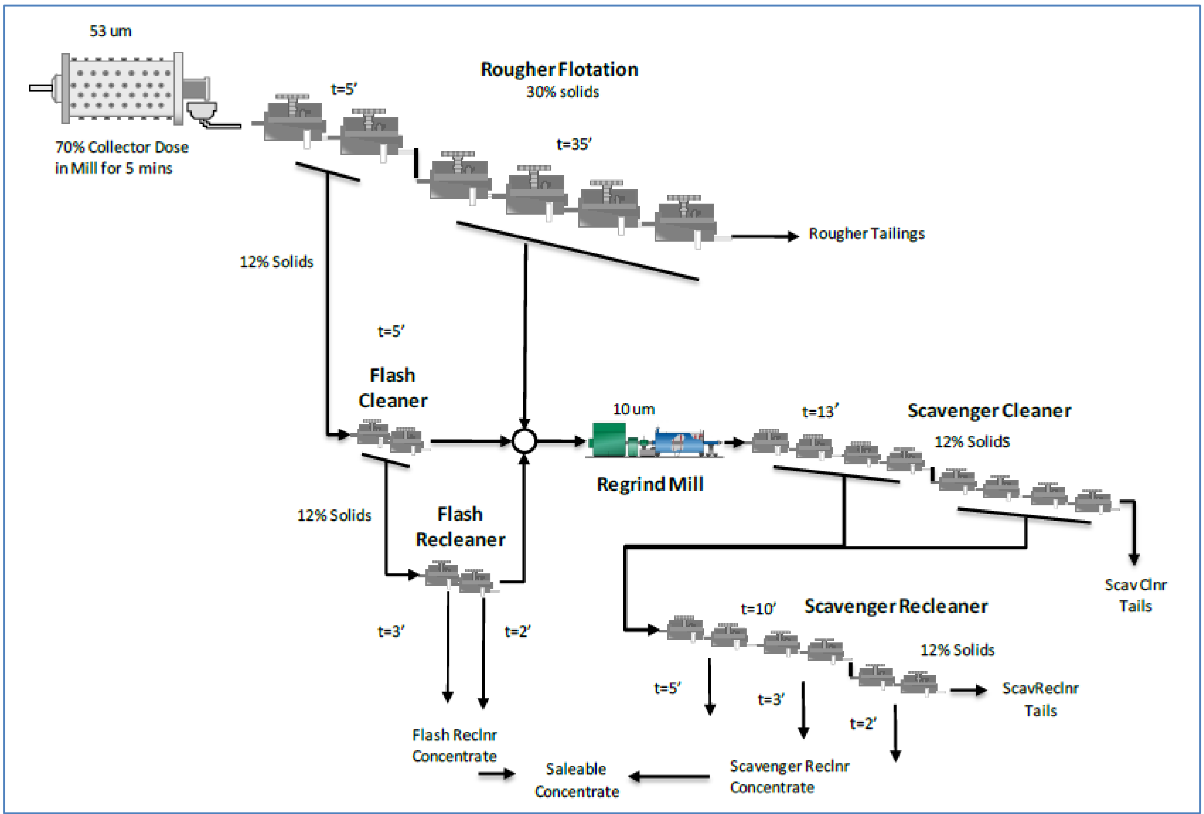


Image courtesy XPS, 2015.

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 (%)	%SiO2	%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
	Hypo	8.48	36.0	86.1	4.00	21.0	0.54	3.54
	Super	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
	Hypo	7.17	35.5	86.9	19.2	13.5	0.41	2.93
	Super	6.02	41.2	65.3	19.3	9.65	1.40	3.80

Note that 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 2016 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

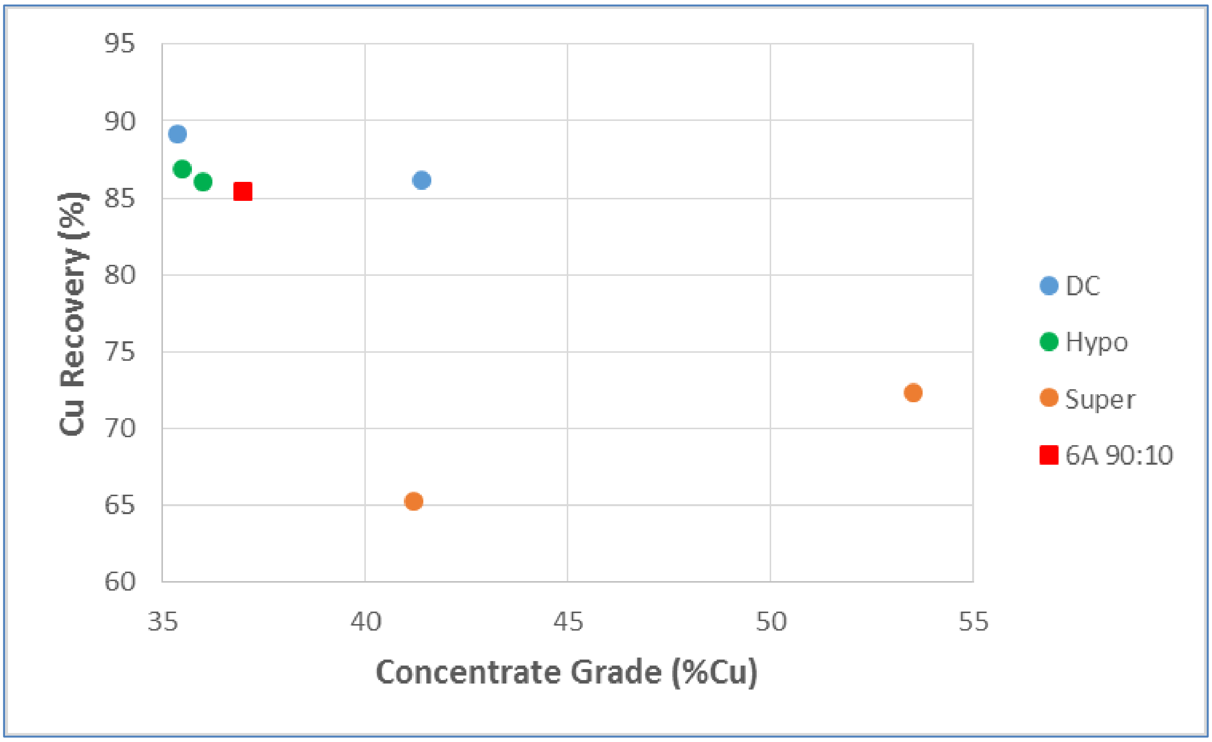


Image courtesy of Amec Foster Wheeler, 2016.

As expected, hypogene ores generate relatively low concentrate grades with good recoveries. The supergene ores 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)

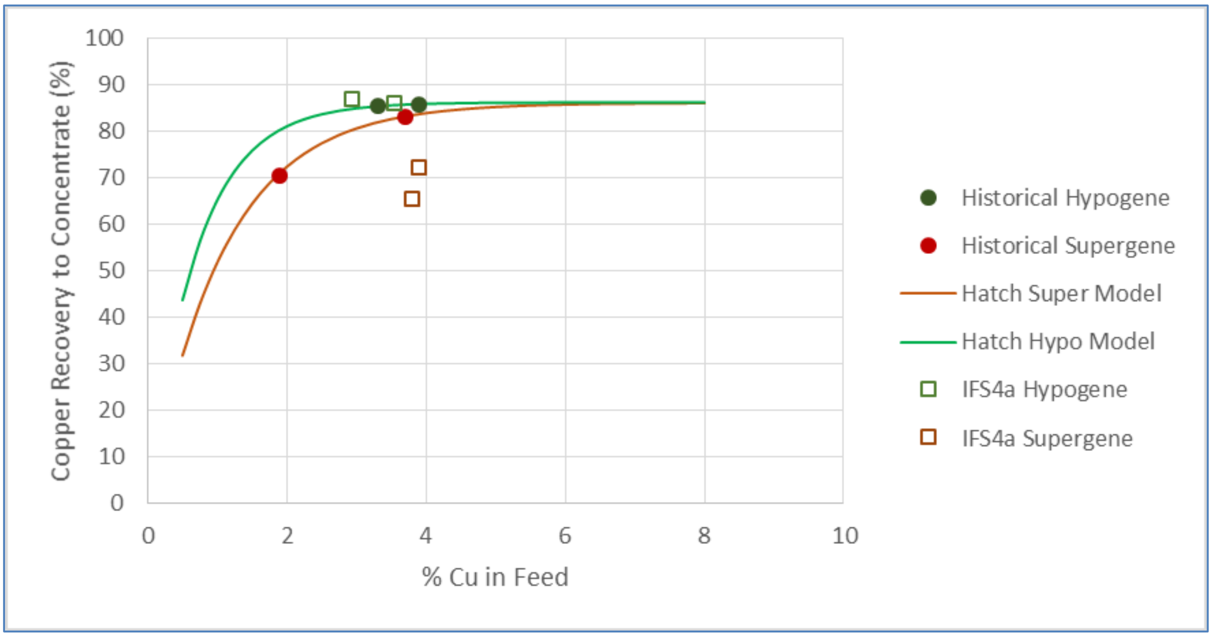


Image courtesy of Amec Foster Wheeler, 2016.

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

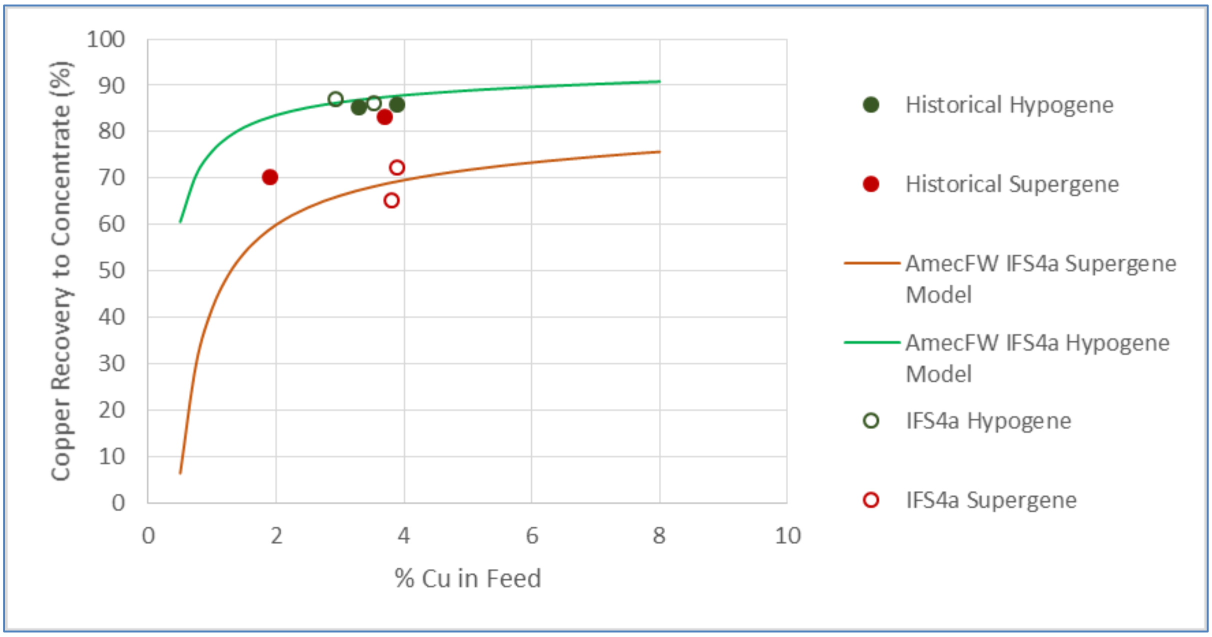


Image courtesy of Amec Foster Wheeler, 2016.

Compared to past models, the new model predicts similar recoveries from hypogene ores and much lower recoveries from supergene ores. 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 2016 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.

Currently the prediction of non-floating copper is not possible from the information in the block model. It is only possible to estimate the non-floating proportion by relying on geological logging of copper mineralisation, conducting actual flotation testing or by quantitative mineralogical analysis. It is recommended that options are explored in the next development phase for geologically defining the non-floating copper proportion throughout the orebody.

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

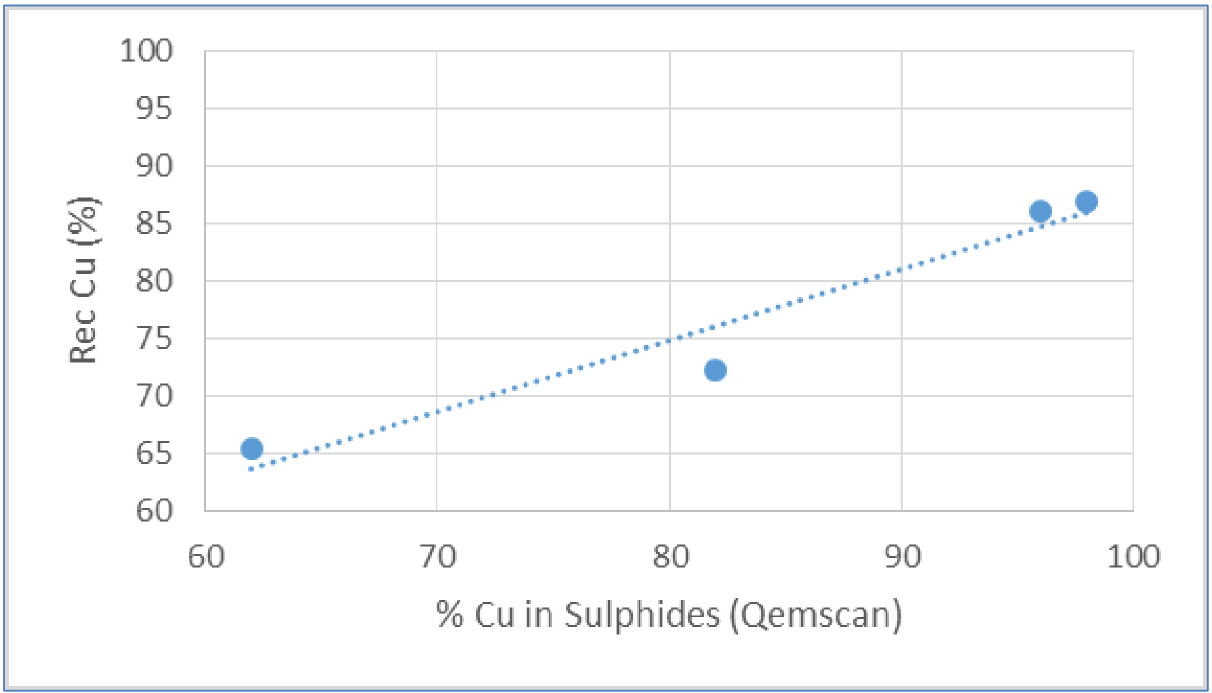


Image courtesy of Amec Foster Wheeler, 2016.

This strong relationship between recoverable copper and copper in sulphides is expected.

13.6.4 Phase 6 Testwork – Signature Plot XPS

A signature plot is used to design IsaMills 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

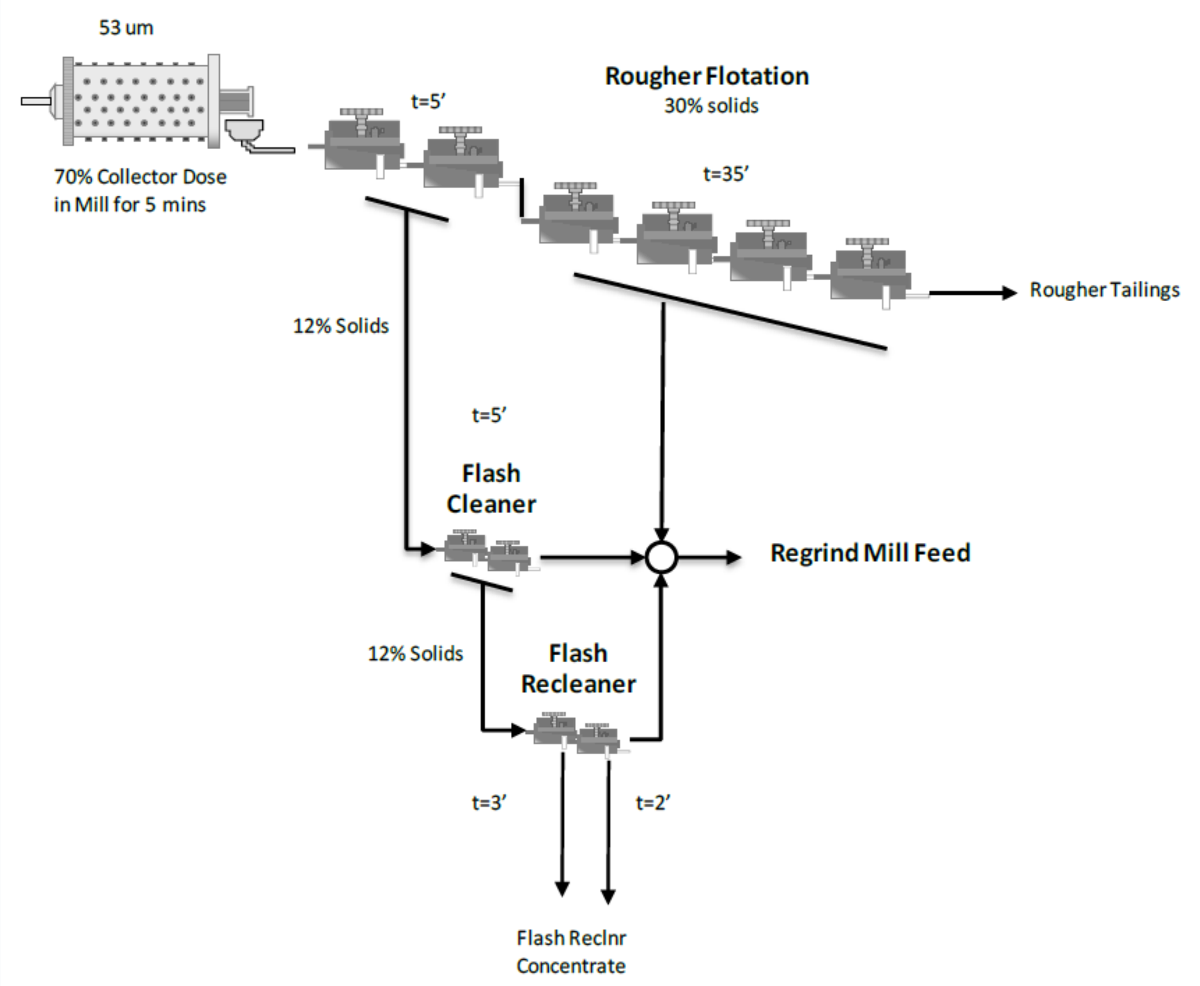


Image courtesy XPS, 2015.

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

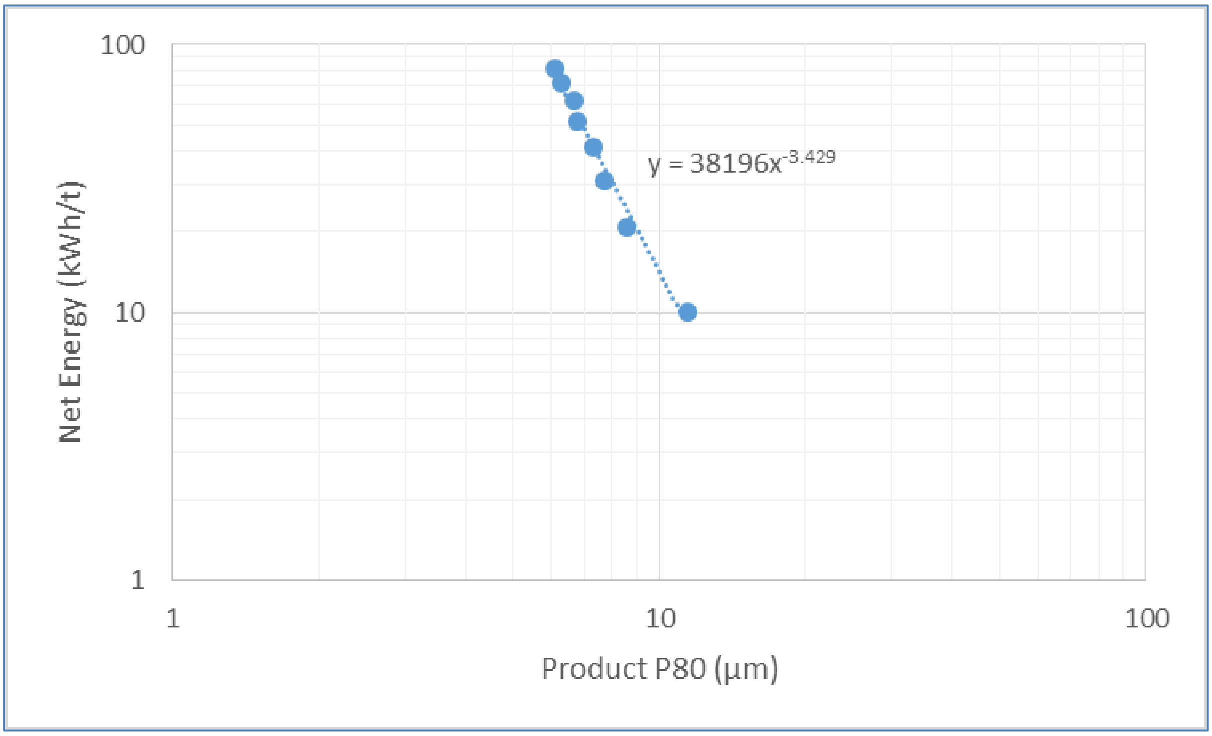


Image courtesy of Amec Foster Wheeler, 2016.

From the starting P_{80} of 34 µm the estimated power to grind to 10 µm is 14.2 kWh/t.

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

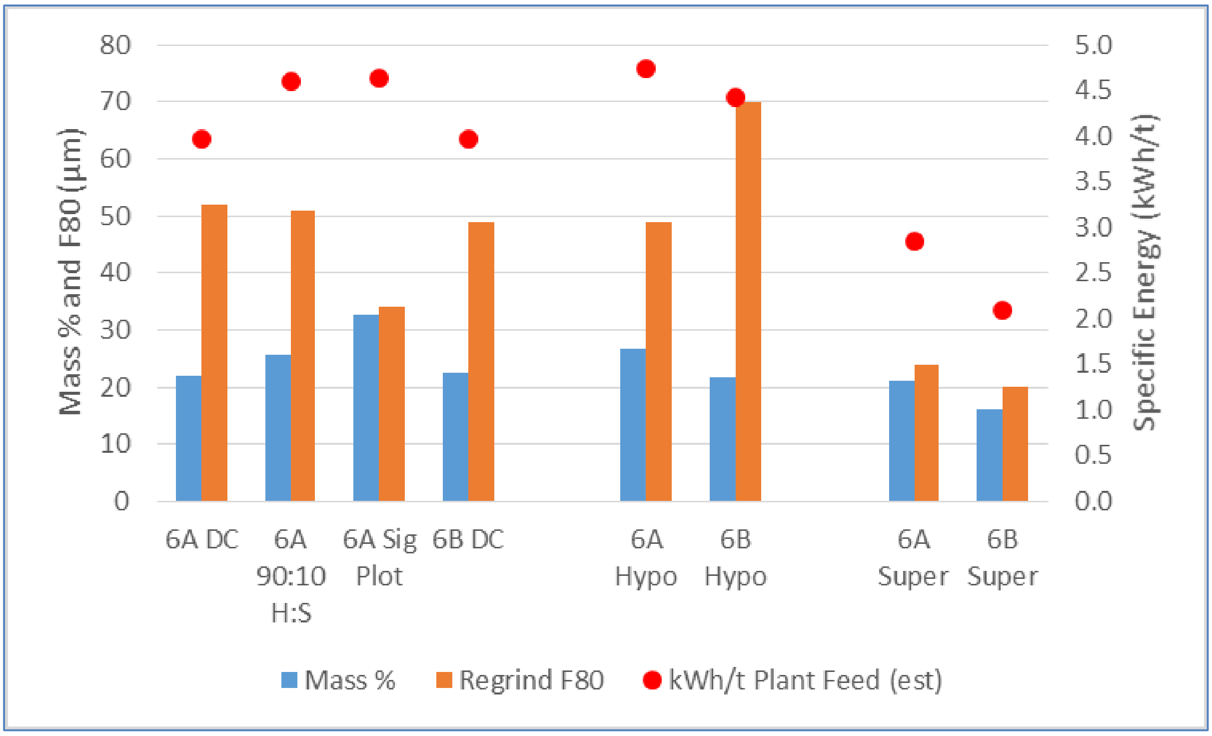


Image courtesy of Amec Foster Wheeler, 2016.

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 2016 PFS circuit. In a 3 Mtpa circuit this equates to a net power requirement of about 2 MW.

The supergene ores only require 3 kWh per tonne of plant feed but are not likely to be processed in isolation.

13.7 Variability Testwork

A program of variability testwork has been planned for Q2' 2016 using the samples indicated in Figure 13.18 together with the year 0 to 15 PFS mining areas.

Figure 13.19 Planned Phase 6 Variability Samples

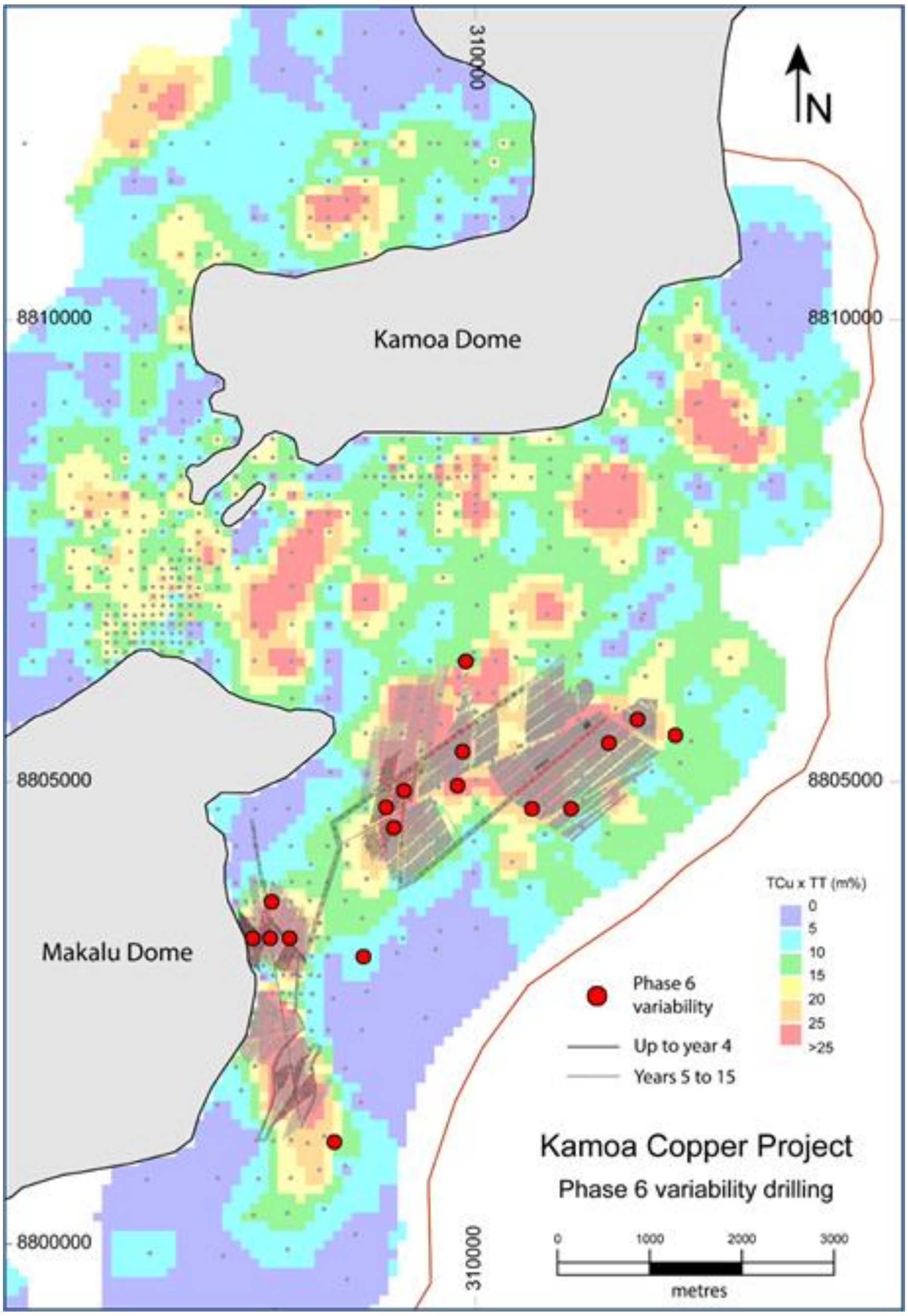


Figure by Ivanhoe, 2016.

Clearly the variability samples will provide a good representation of plant feed during the proposed mine plan period. The variability results, when available, will be analysed to understand metallurgical response by mine area and by ore delivery timing.

13.8 Process Mineralogy

The Kamoa copper sulphide mineralisation exists in two basic modes regardless of copper sulphide mineral. Coarse copper sulphides, some in the centimetre size range, are clearly visible in the core. Many intermediate sized copper mineral grains are usually visible but any that are clearly distinguishable can be considered coarse. The second mode of occurrence is a pervasive "fog" of ultrafine copper sulphides throughout the matrix.

In the image below (Figure 13.20) can be seen a 2 cm wide white clast within the grey diamictite matrix, against which chalcopyrite has "mantled" during the sulphide deposition phase. In the surrounding rock matrix there are smaller mantled clasts and visible blebs of chalcopyrite (and other sulphides). What cannot be seen in the photographis the dispersion of 1 to 10 μm (0.0001 to 0.001 cm) copper sulphides present throughout the grey matrix.

Figure 13.20 Typical Kamoa Hypogene Mineralisation in Diamictite

Figure Courtesy Amec Foster Wheeler, 2011.

QEMScan, an automated particle analysis system, has been used to reveal the fine mineralogical detail of Kamoa samples. Two rougher flotation tests were conducted the 6A development composite by XPS in which six concentrates were collected sequentially after grinding the ore to P_{80} 53 µm and 38 µm respectively. The QEMScan analysis was used to derive the proportion of liberated copper in each of the concentrates and the results are summarised in Figure 13.21.

Figure 13.21 Copper Sulphide Liberation in Rougher Flotation

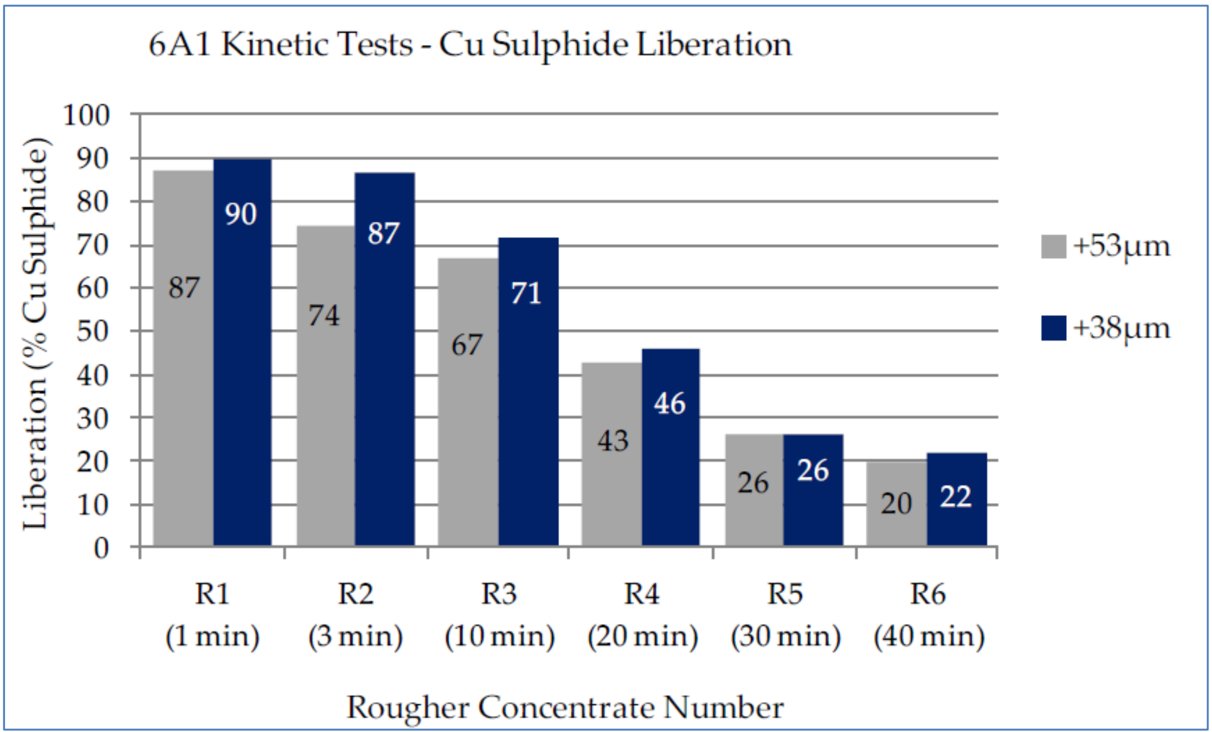


Image courtesy XPS, 2015.

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

QEMScan also generates particle mineral maps and is able to group both minerals and particles to assist in visual examination. Figure 13.22 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 so the definition of liberation is based on a further grouping of these two classes.

Figure 13.22 Combined Copper Sulphides Liberation Map- Rougher Concentrates R3 to R6

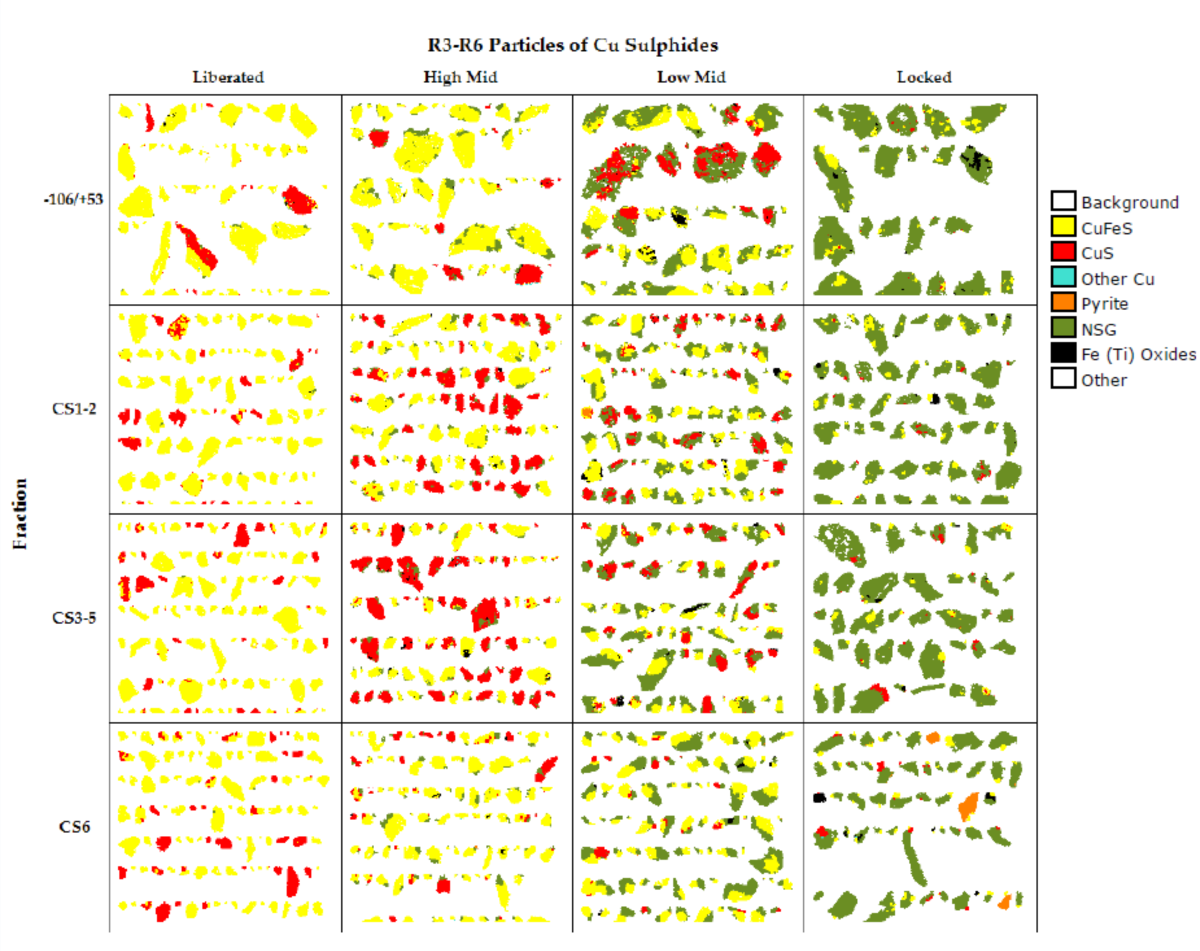


Image courtesy XPS, 2015.

It is clear that even in the CS6 (cyclosizer cone 6, 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 flowsheet development.

All particles in Figure 13.22 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 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 to 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 13.22 above is the general absence of pyrite. It is only at the finest size that pyrite appears and this indicates that composites between pyrite and copper sulphides are scarce.

The copper that is lost in flotation has been examined using QEMScan analysis of the rougher tailings. The liberation map for Rougher tails is shown in Figure 13.23.

Figure 13.23 Combined Copper Sulphides Liberation Map - Rougher Tails

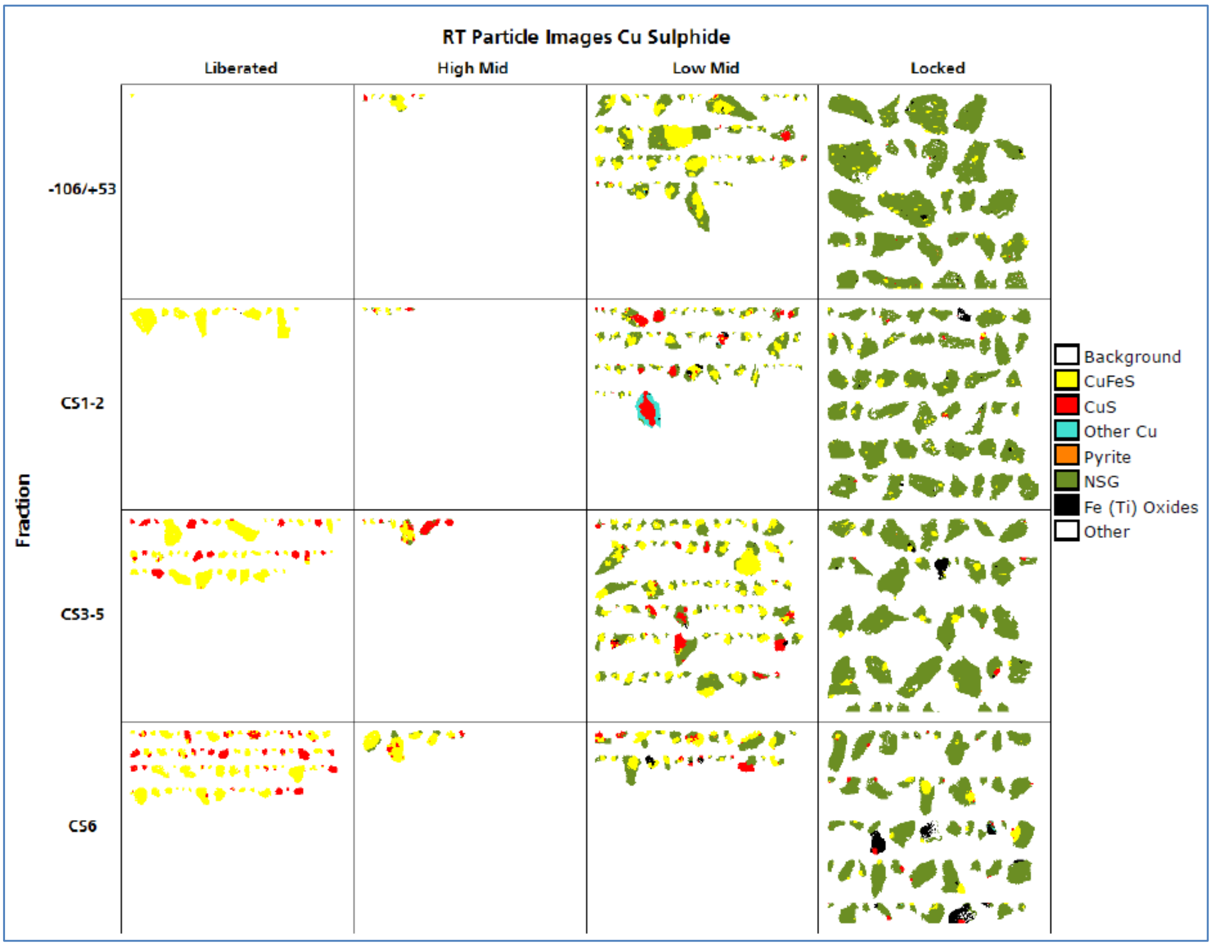


Image courtesy XPS, 2015.

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 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 form of mineralisation in each in three dimensions is unknown). As can be seen in Figure 13.24, regardless of the size fraction the lost copper sulphides are in phases that are sub 10 µm.

Figure 13.24 Copper Sulphide Phase Size in Rougher Tailings

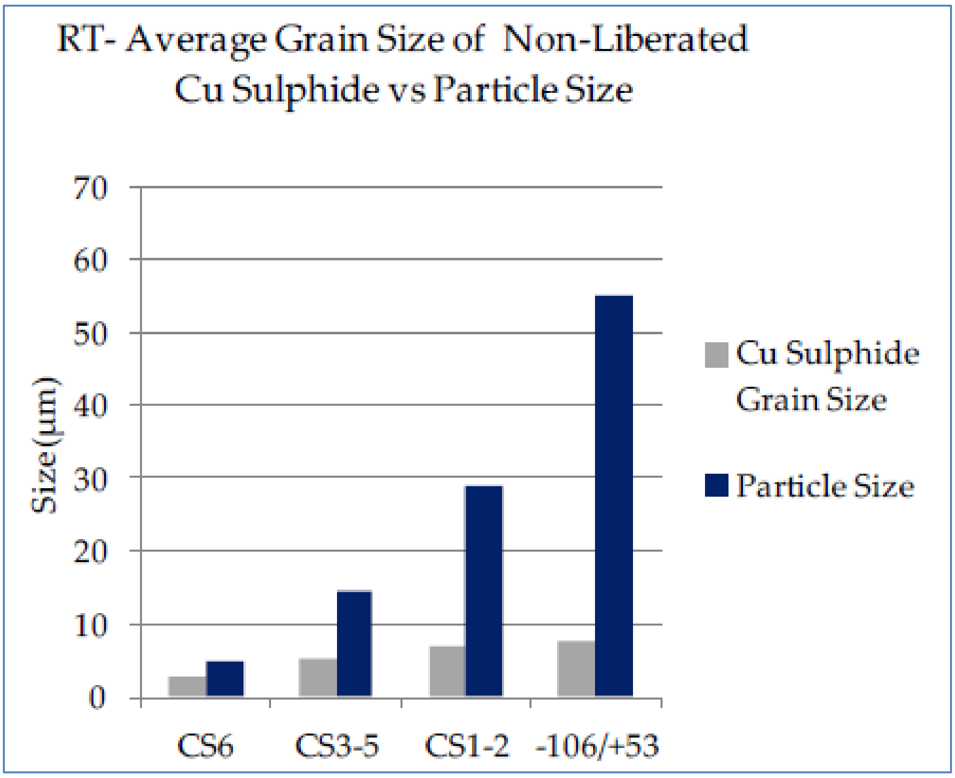


Image courtesy XPS, 2015

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 established and high recoveries are being achieved at high concentrate grades.

13.9 Comments on Section 13

In the opinion of the Amec Foster Wheeler QP the metallurgical testwork is sufficient for PFS level process design for the Kamoa Project. The comminution characteristics are well established and have consistency across the various testing phases. The flotation characteristics are well understood and explainable in terms of the process mineralogy. The samples tested in Phase 6 represent the material to be mined in processed according to the Kamoa 2016 PFS schedule. There are some remaining uncertainties in terms of the crushing properties of the minzone and the variability in properties of the footwall and hangingwalls.

The flotation characteristics are well understood and explainable in terms of the process mineralogy. The samples tested in Phase 6 represent the material to be mined in processed according to the Kamoa 2016 PFS schedule. The remaining uncertainty is the level of true representation provided by the supergene samples with both Phase 6 composites having historically high levels of non-floating copper minerals which have resulted in low copper recoveries.The ore does not contain deleterious elements often found in copper concentrates such as arsenic or fluorine. 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 recoveryof silica through attachment with these sulphides. This, in turn, has led to silica rejection issues in final concentrate production, which is mitigated to some degree by 10 µm regrinding. The remaining silica recovery issue relates to consistency of rejection with Phase 6 final concentrates ranging in silica content from 4% to 19% using the same flowsheet on different samples. This broad range has the potential to be problematic for customers and investigations are required into methods of silica content control.

The prediction of copper recovery from hypogene ore is reasonable but the prediction for supergene samples is currently inadequate. The proposed supergene recovery prediction curve in Figure 13.14 is felt to be conservative because is driven by historically high levels of non-floating copper minerals in what are thought to be representative Phase 6 supergene samples. An improved method of supergene recovery prediction is necessary for the next phase of project development.

The power required to conduct regrinding has been estimated for one sample (using an IsaMill signature plot) and been estimated to be reasonably consistent across the Phase 6 tests.

One characteristic of the Kamoa Project has been the changing mine locations through the phases. The work performed to date is appropriate for the Kamoa 2016 PFS mine plan but will not be adequate should the mine plan change to incorporate significant mineralisation that have not yet been tested for either comminution or flotation response.

14 MINERAL RESOURCE ESTIMATES

14.1 Key Assumptions/Basis of Estimate

The underground resource models discussed in this section were constructed by Mr. George Gilchrist, Pr. Sci Nat, Ivanhoe's Mineral Resources Manager. The models are described in Gilchrist (2014) and were reviewed and verified by Amec Foster Wheeler (Seibel, 2014). The 2014 Mineral Resource was estimated using the same approach adopted in 2013, with minor variations in the application of grade capping and the use of twin or wedge holes. The effective date of the Mineral Resource estimates is 5 May 2014. The qualified persons for the estimates are Dr Harry Parker, RM SME, and Mr Gordon Seibel, RM SME, employees of Amec Foster Wheeler.

The models were created using criteria that assume the deposit would be exploited as an underground mining operation. Unlike the November 2013 PEA, no open-pit models were constructed, as the Kamoa 2016 PFS focus is on underground mining.

Amec Foster Wheeler considers the Mineral Resource models and Mineral Resource estimates derived from those models to be consistent with Canadian Institute of Mining Metallurgy and Petroleum (CIM) 2014 Definition Standards (2014 CIM Definition Standards) and the 2003 CIM Best Practice Guidelines for Mineral Resource and Mineral Reserve estimation (CIM, 2003).

14.2 Selective Mineralized Zones (SMZ)

The Mineral Resource estimate used 720 drillhole intercepts, which include drillholes within the mining lease, but excludes drillholes on the Kamoa and Makalu domes; these are areas where the Ki1.1.1 is not present, or where the mineralisation has been completely leached. Included in the 720 drillholes are 16 twin holes (where the spacing between drillholes is <25 m) and six wedge holes. These drillholes were used in the estimation, but were given a cumulative weighting to prevent estimation biases due to clustering.

Collar, survey, assay, lithology and density 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 drillhole file was exported to Excel, and the best single selected mineralized SMZ selections in each hole were manually selected.

The best single selected mineralized intercept (SMZ10) field was selected and flagged by hand if the following criteria were met:

- Approximate minimum downhole length of 3 m.
- Total copper (TCu) grades are greater than 1% TCu.

If the best composite was slightly less than 3 m, and a long waste interval was required to meet the minimum 3 m guideline, then an SMZ with a length slightly less than 3 m was accepted to avoid unnecessary dilution. In the event that a drillhole did not meet the minimum grade ≥1% TCu and length greater than 3 m criteria, the highest-grade 3 m composite was formed in the appropriate stratigraphic position. These composites were included in the resource estimations to define the geometry of the SMZ, and to introduce a degree of lateral dilution into the model.

However, in a few cases, two composites could be formed within a single drillhole at different elevations. In these cases, only the composite consistent with the stratigraphic position of neighbouring composites was used. Amec Foster Wheeler independently constructed the SMZ selections, reviewed the differences with Ivanhoe, and found that any differences noted were non-material.

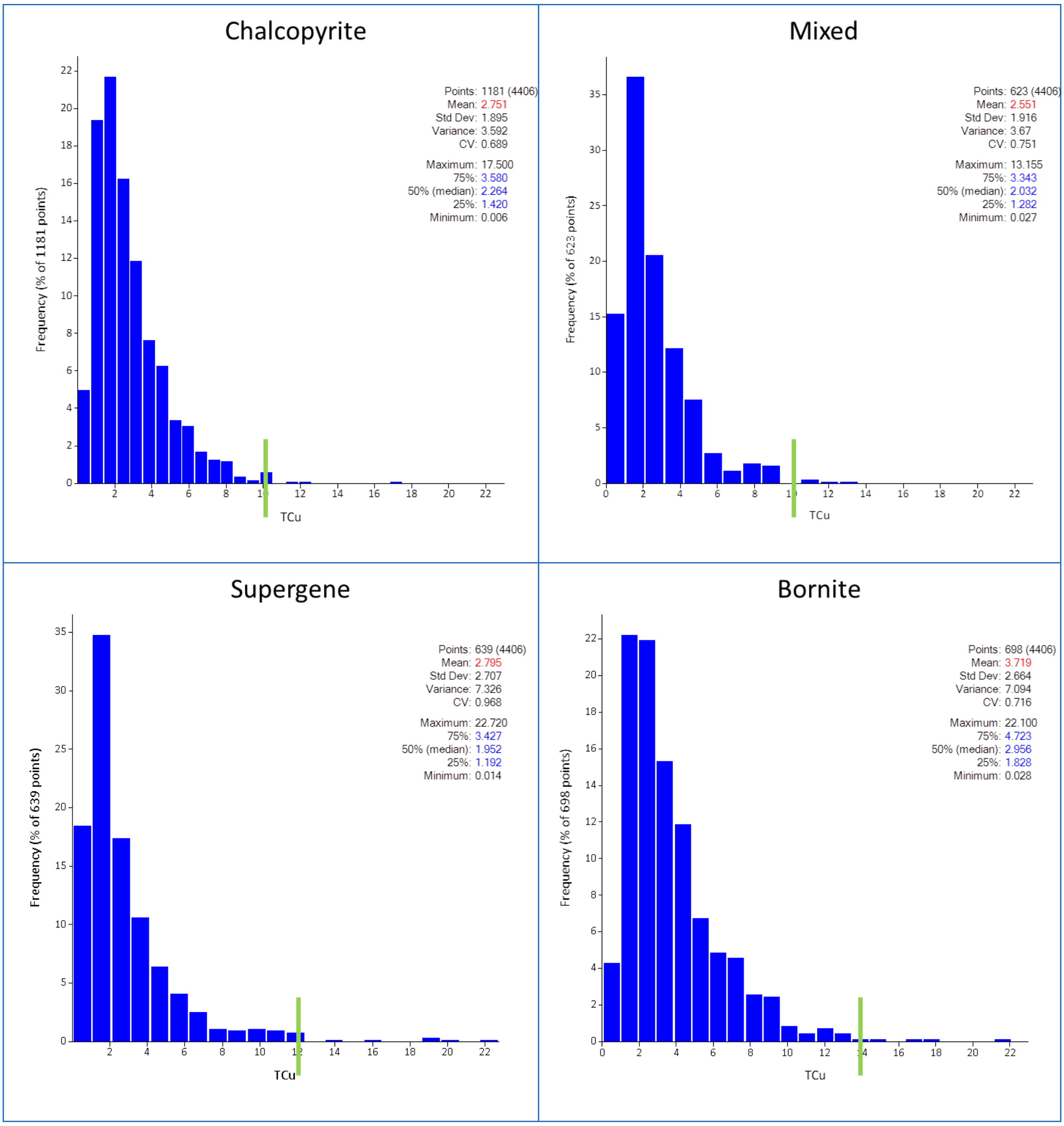
The assay *.csv file, with the SMZ selections flagged, was imported into Datamine® Studio mining software where it was combined with the collar and survey files, and composited into a single SMZ for each hole using the SMZ flagged field. All TCu composites were then capped at 10% except for the bornite zone which was capped at 14% and the supergene zone that was capped at 12%. The effect of the capping is considered minimal as the overall grade of the TCu composites was decreased by 0.7%. The acid soluble copper (ASCu) grade of the composites was set to the TCu grade if the ASCu grade was greater than the corresponding TCu grade.

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 SMZ, and a default value of '0' for all other samples. The drillhole was composited over the full SMZ interval (where the SMZ selection field equalled '1') forming a single composite for each drillhole with intersection thickness and average grade.

Top Capping

All samples within the SMZ were composited to approximate 1 m intervals prior to capping to ensure equal sample support. Histograms and log probability plots were investigated per mineral zone, with a capping grade selected based on the grade at which the population distribution began to break down, refer to Figure 14.1.

Figure 14.1 Example Histograms of Top Capping Selection for a Number of the Mineral Zones



Although the coefficient of variation (a measure of relative variability determined by dividing the standard deviation by the mean, CV) for all zones is <1.0 (considered to be low), isolated high grade samples can have too much influence during estimation. A top capping grade of 10% TCu was thus selected for all zones, with the exception of the higher-grade supergene and bornite zones, where capping grades of 12% TCU and 14% TCu respectively were applied. The capping was applied to the 1 m composite samples per mineral zone prior to creating a full width composite across each of the SMZ intervals. Table 14.1 shows the impact of top capping.

Table 14.1 Impact of Top Capping Per Mineral Zone on 1 m Composite Samples (SMZ10 Option)

Mineral Zone	Number of samples	Capping grade TCu (%)	Samples capped	No capping		With capping	
				Mean (%)	CV	Mean (%)	CV
Bornite	698	14%	5	3.72	0.72	3.70	0.69
Chalcopyrite	1181	10%	6	2.75	0.69	2.74	0.67
Cpy-Bn-Cct	447	10%	2	2.56	0.67	2.56	0.65
Mixed	623	10%	4	2.55	0.75	2.54	0.74
Supergene	639	12%	7	2.80	0.97	2.74	0.88
Py>>Cpy	76	10%	0	0.90	0.84	0.90	0.84

14.3 Exploratory Data Analysis

Composite statistics for the base case 1.0% TCu cut-off SMZ (SMZ10) are summarized in Table 14.2, and displayed graphically as histograms and log probability plots in Figure 14.2 through Figure 14.6. The relationships between total copper and true thickness were evaluated using scatter plots in Figure 14.7.

Table 14.2 Composite Statistics for each SMZ option (Capped Data)

Variable	SMZ	Number of samples	Minimum	Maximum	Mean	Standard Deviation	CV
TCu (%)	SMZ10	720	0.01	10.54	2.70	1.38	0.51
ASCu (%)	SMZ10	666	0.001	3.61	0.30	0.32	1.05
True Thickness (m)	SMZ10	720	2.00	17.91	5.53	2.85	0.52
TCu x TT	SMZ10	720	0.04	103.09	14.92	11.99	0.80

Note: TCu, ASCu statistics weighted by true thickness.

Figure 14.2 Capped Composite TCu (%) for SMZ10 (1.0% TCu cut-off)

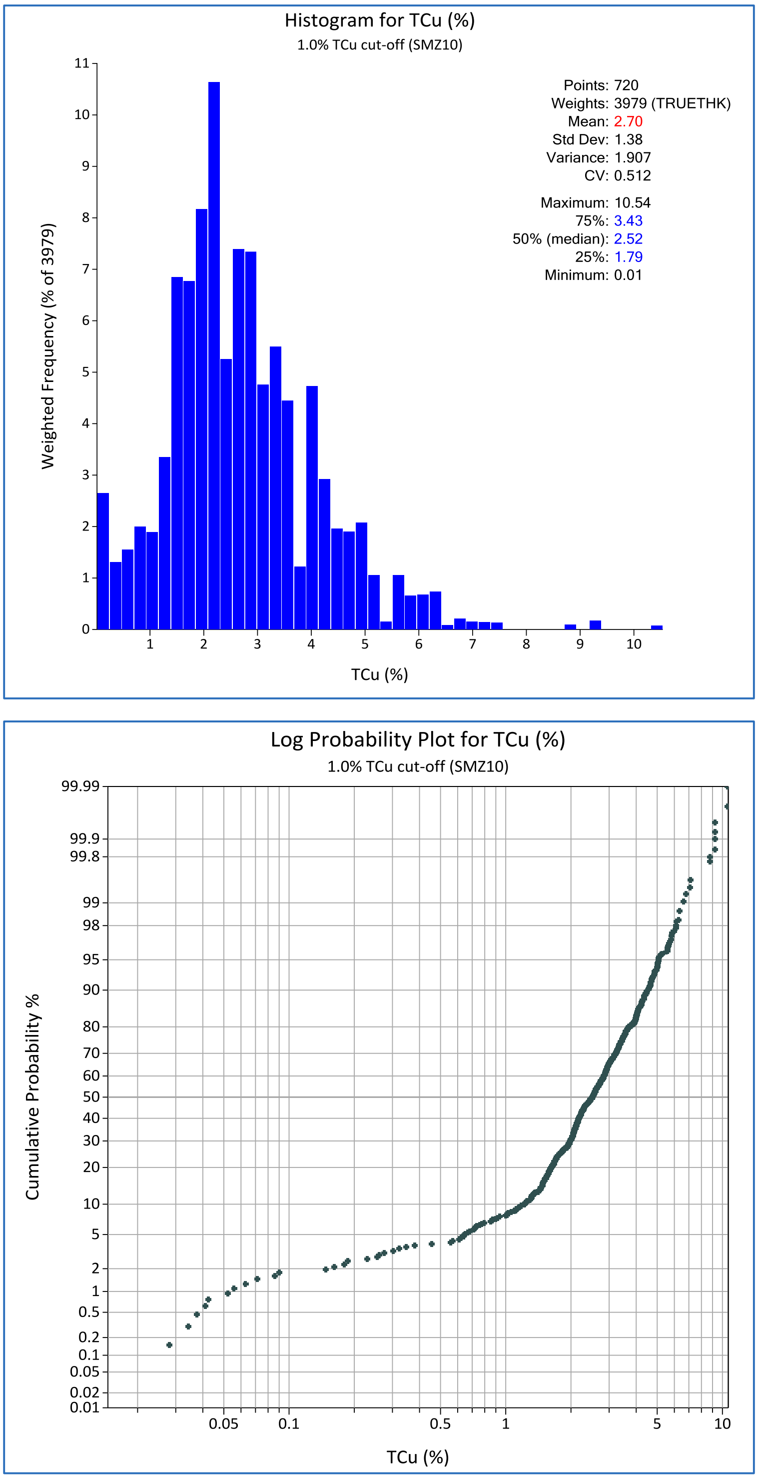
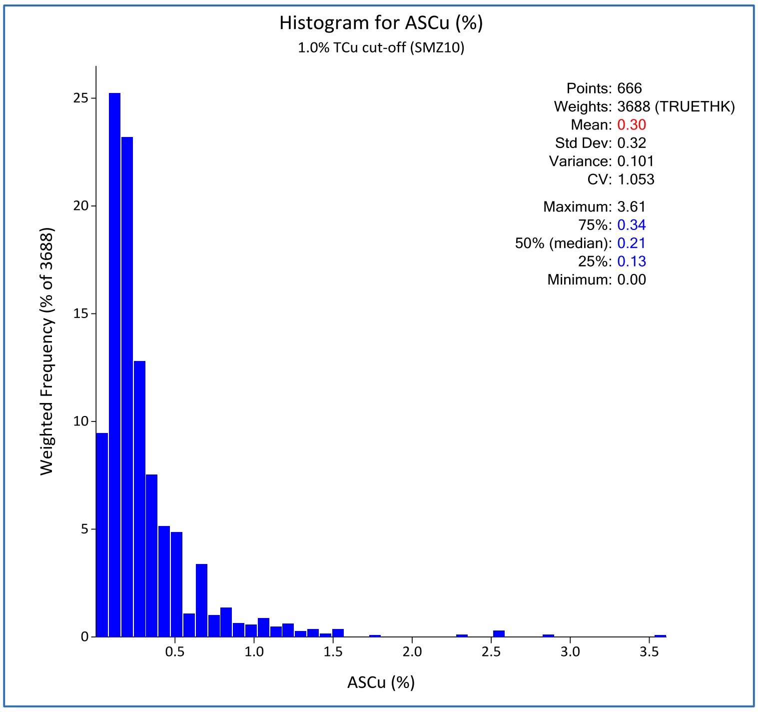


Figure provided by Ivanhoe, 2014; histogram and probability plot (weighted by true thickness).

Figure 14.3 Capped Composite ASCu (%) for SMZ10 (1.0% TCu Cut-off)



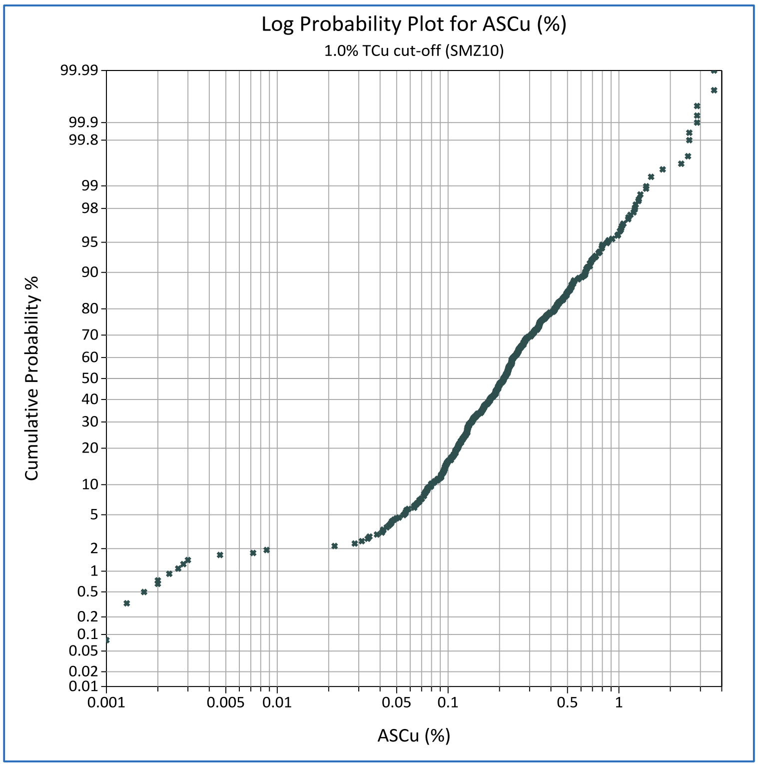
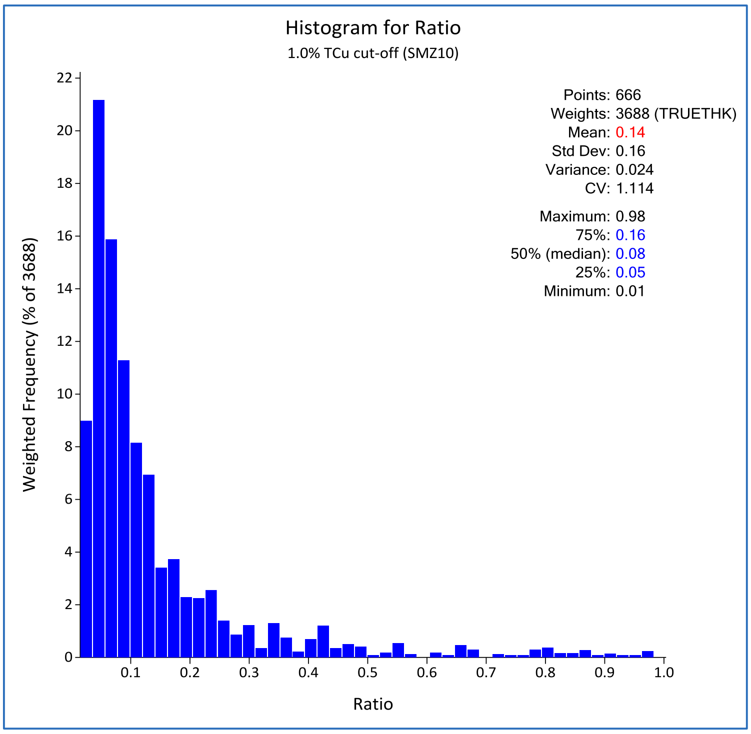


Figure provided by Ivanhoe 2014; histogram and probability plot (weighted by true thickness).

Figure 14.4 Capped Composite TCu:ASCu Ratio for SMZ10 (1.0% TCu cut-off)



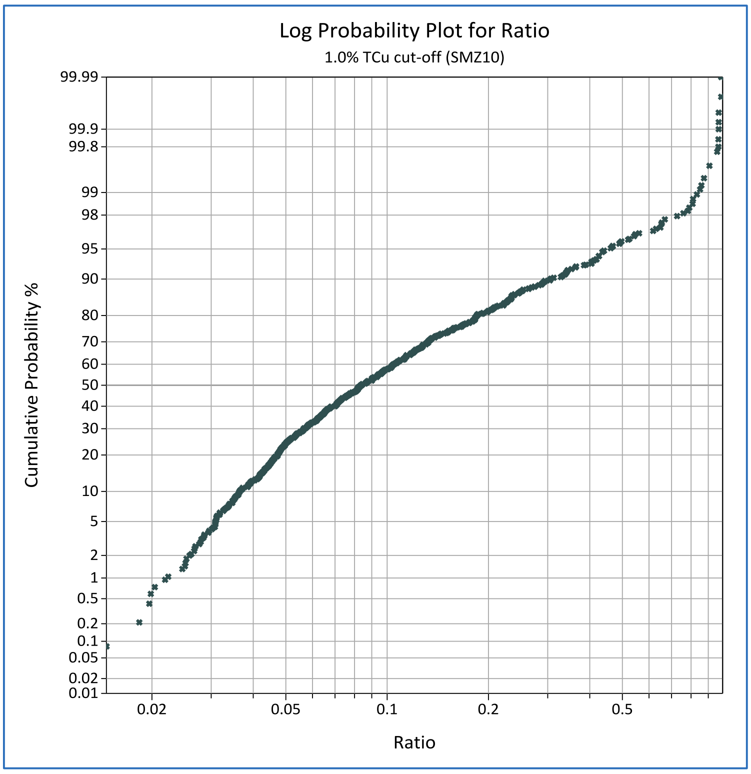
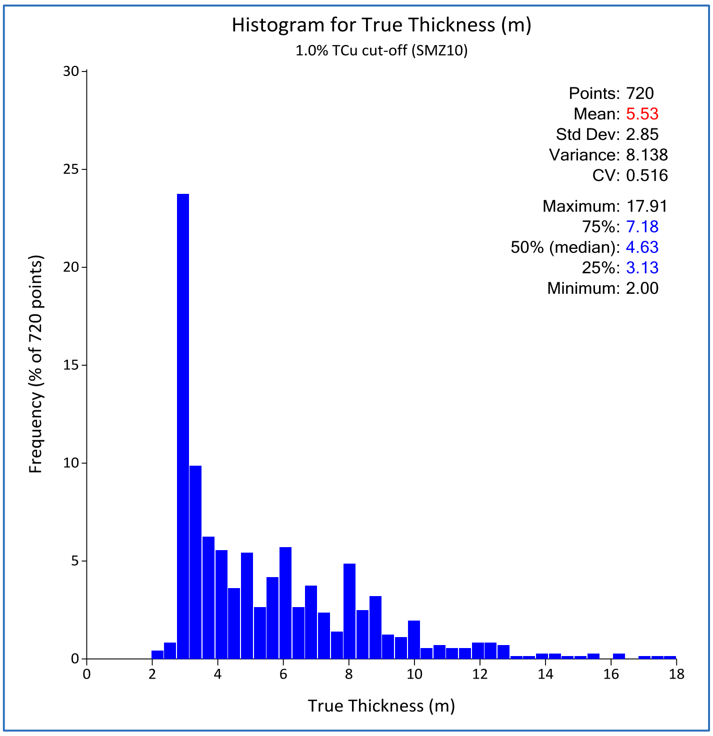


Figure provided by Ivanhoe, 2014; histogram and probability plot (weighted by true thickness).

Figure 14.5 Capped Composite True Thickness for SMZ10 (1.0% TCu Cut-off), Histogram and Probability Plot



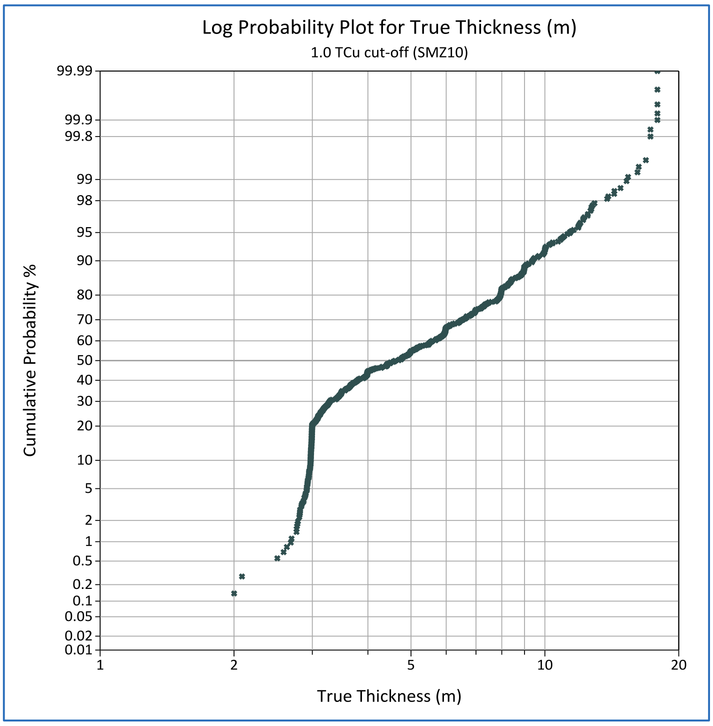
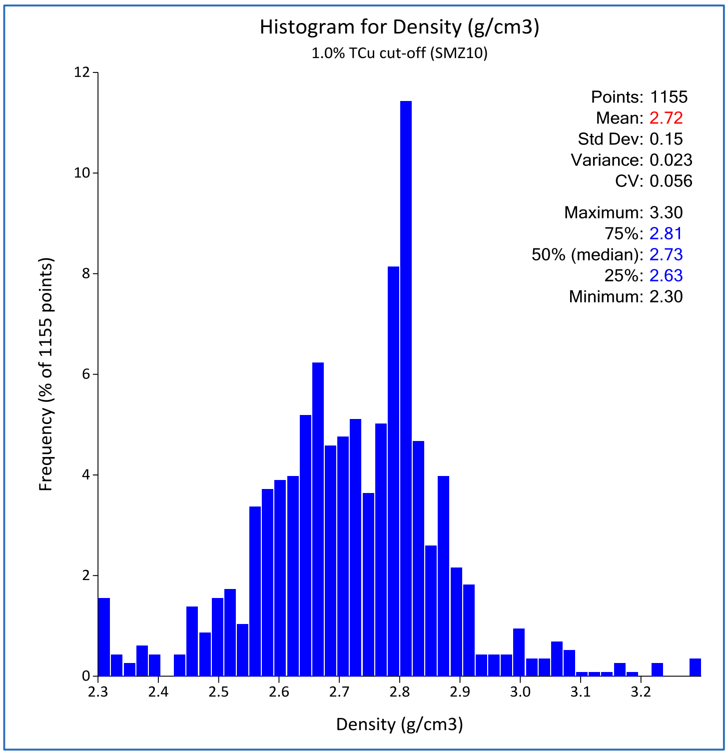


Figure prepared by Ivanhoe 2014; histogram and probability plot (weighted by true thickness).

Figure 14.6 Density for SMZ10 (1.0% TCu Cut-off)



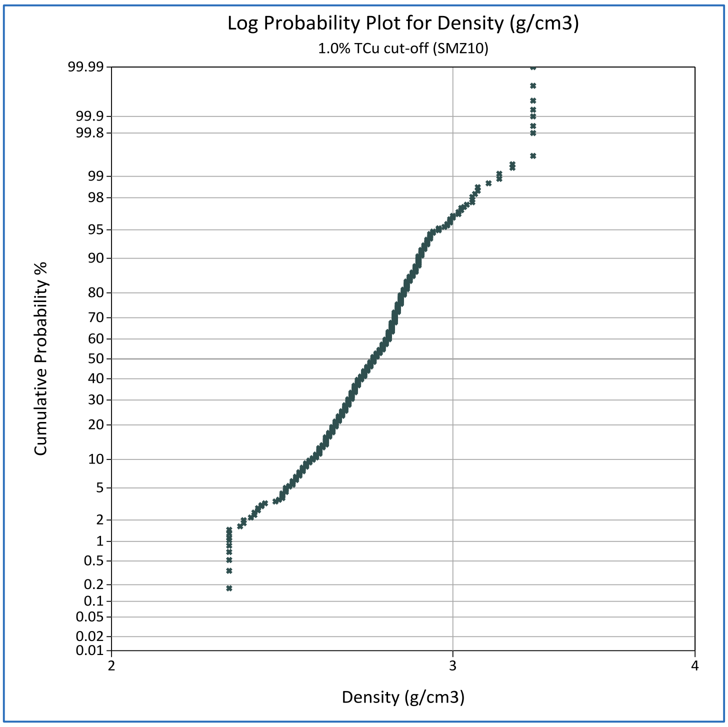
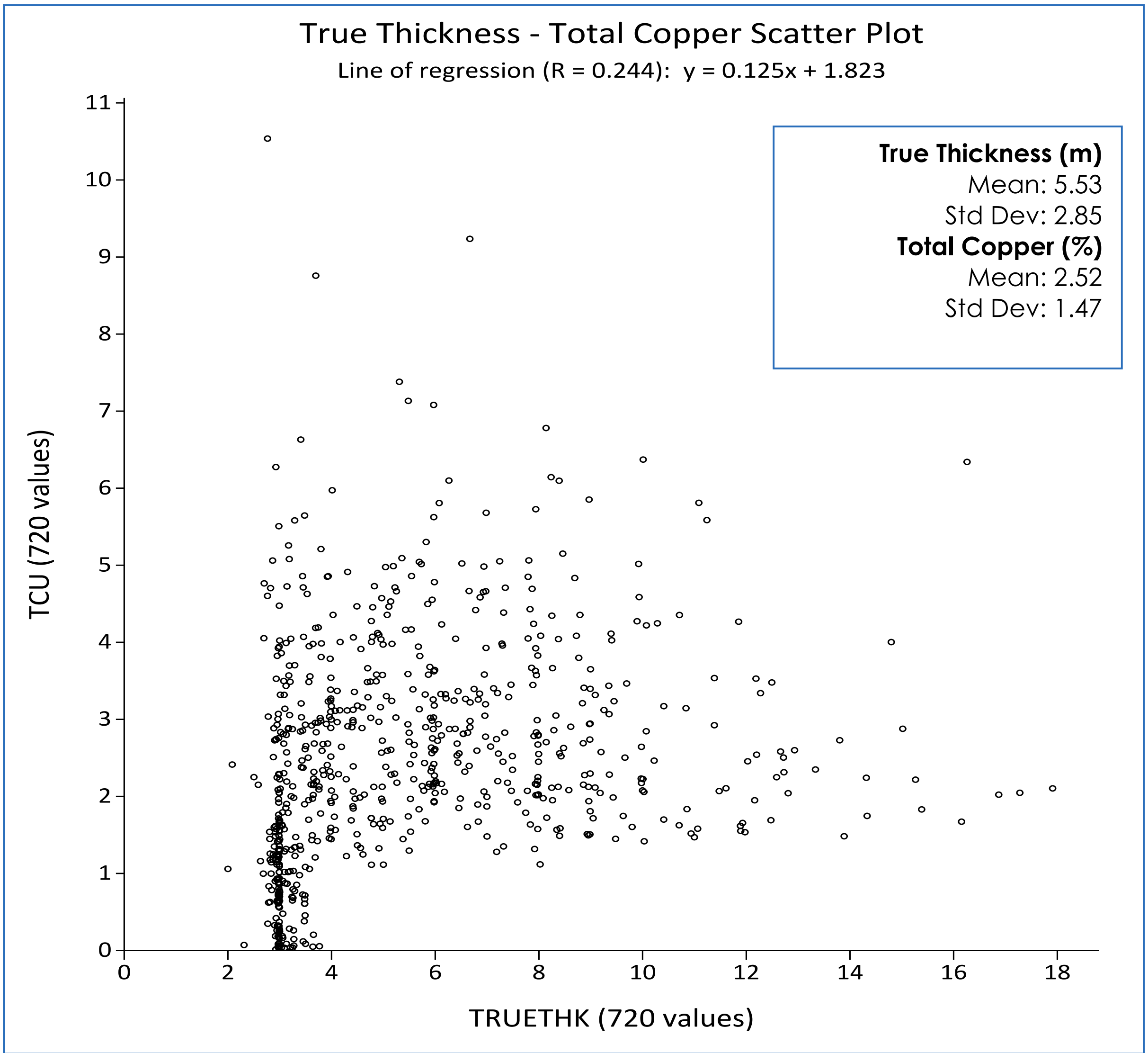


Figure prepared by Ivanhoe 2014; histogram and probability plot (weighted by true thickness).

Figure 14.7 Scatter Plot of Total Copper (%) Versus True Thickness (m)



Note: Figure prepared by Ivanhoe 2014.

14.4 Statistics Observations

Total copper grades are well constrained and approximate a normal distribution. Although the SMZ is defined above a 1.0% cut-off, a number of drillholes do not have sufficient mineralisation to meet that criterion. In such cases, the highest grade 3 m interval was selected to ensure the lower-grade zones are represented in the model, and provide lateral dilution during estimation. The lower-grade drillholes are typically on the edges of the drilled area.

ASCu and the ratio between ASCu/TCu (the Ratio) are positively skewed, approximating a log-normal distribution. This is supported by geological and metallurgical studies of the sulphide species that indicate that the bulk of the mineralization at Kamoa is sulphide, with localized oxide mineralization closer to surface and along dome edges. No significant inflection in the log probability plot is evident, indicating an inability to distinguish sulphide and oxide mineralization into separate domains at this stage using the Ratio.

True thickness is positively skewed from a modal value of 3 m (indicating the minimum thickness coded). A few samples below the minimum 3 m are a function of the correction from drilled thickness (on which the SMZ is based) to true thickness. Density values show very little variability, with a CV of 0.06.

The scatter plot between total copper and true thickness indicates a very weak relationship between the two. Although statistically not necessary to estimate copper weighted by true thickness, the weighting approach was adopted in the grade estimation in case local stronger correlations exist.

14.5 Structural Model

Though the mineral resource estimation is performed in 2D, to support mine planning a 3D model is required. Thus it was necessary to make a structural model to enable the elevation of the SMZs to be estimated.

SRK identified eight major structures using geophysical data, and lithological discontinuities interpreted from the drillhole data Section 7.3.5). These structures were then used to divide the model into nine structural zones with internally similar strikes and dips. For grade estimations, the model and SMZs were flattened to two dimensions (2D), with the SMZs allowed to be included across the structural domain boundaries. Amec Foster Wheeler concurs that this approach appears reasonable, as the faulting in most of Kamoa appears to have occurred after the deposition of the mineralization.

At this time, it is difficult to establish the dips of these faults, and/or to determine if they are a single fault plane or represent a fault zone. For the resource model, the simplest interpretation of the faults was used, which assumed that the faults are single vertical planes. 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. Recent structural information from the exploration drift currently in progress 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 mineralization and its implication on the efficacy of the proposed mining methods and Project economics.

14.6 Copper Resource Model

Surface modelling and block model creation were limited to within perimeters defining the mineralized portions and permit boundaries of the Project. Two prominent domes, the Kamoa Dome to the north and the Makalu Dome to the south, were excluded from the modelling as they represent leached areas, or barren areas where the Roan sandstone (R4.2) crops out at surface. The outer extents of the bounding perimeters were defined by the mining permit, or an outline defining the outermost extents of drilling.

The Mineral Resource area was subdivided into nine structural domains based on the revised structural model. A digital terrain model (DTM) was created through the centroid of each of the SMZ intervals to define the geometry of the SMZ within individual fault blocks. To avoid anomalies in the surfacing, control lines were used along the fault surfaces to guide the DTM creation.

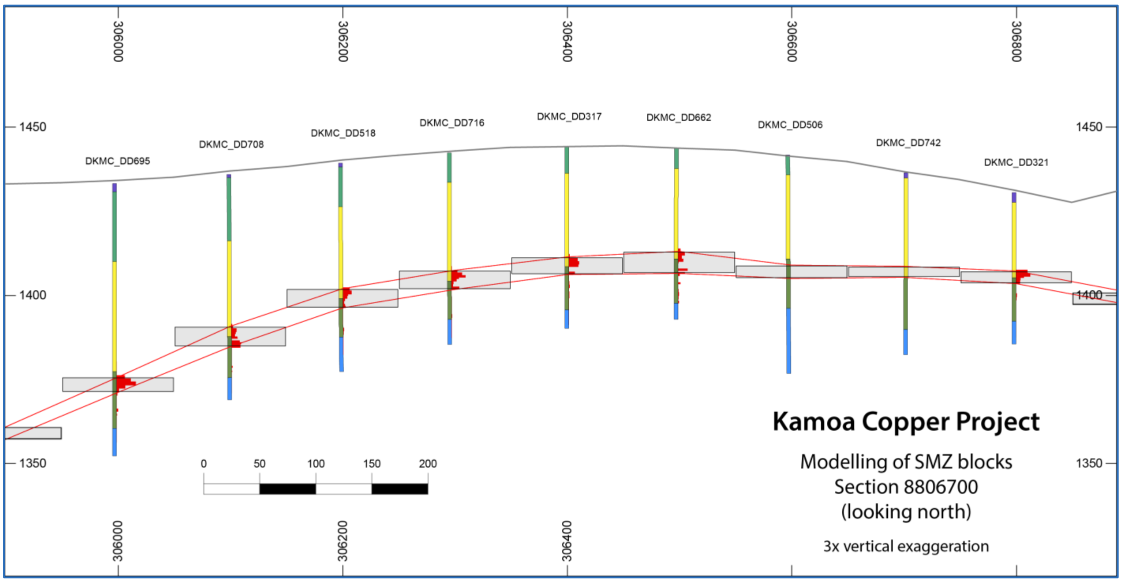
A gridded seam model was established using 100 m by 100 m blocks in easting and northing, with a single block in elevation using the parameters in Table 14.3.

Table 14.3 Block Model Parameters

Axis	Origin	Maximum	Block Size (m)	# Blocks
Easting (X)	295,050	320,050	100	250
Northing (Y)	8,797,050	8,827,050	100	300
Elevation (Z)	N/A	N/A	Variable	1

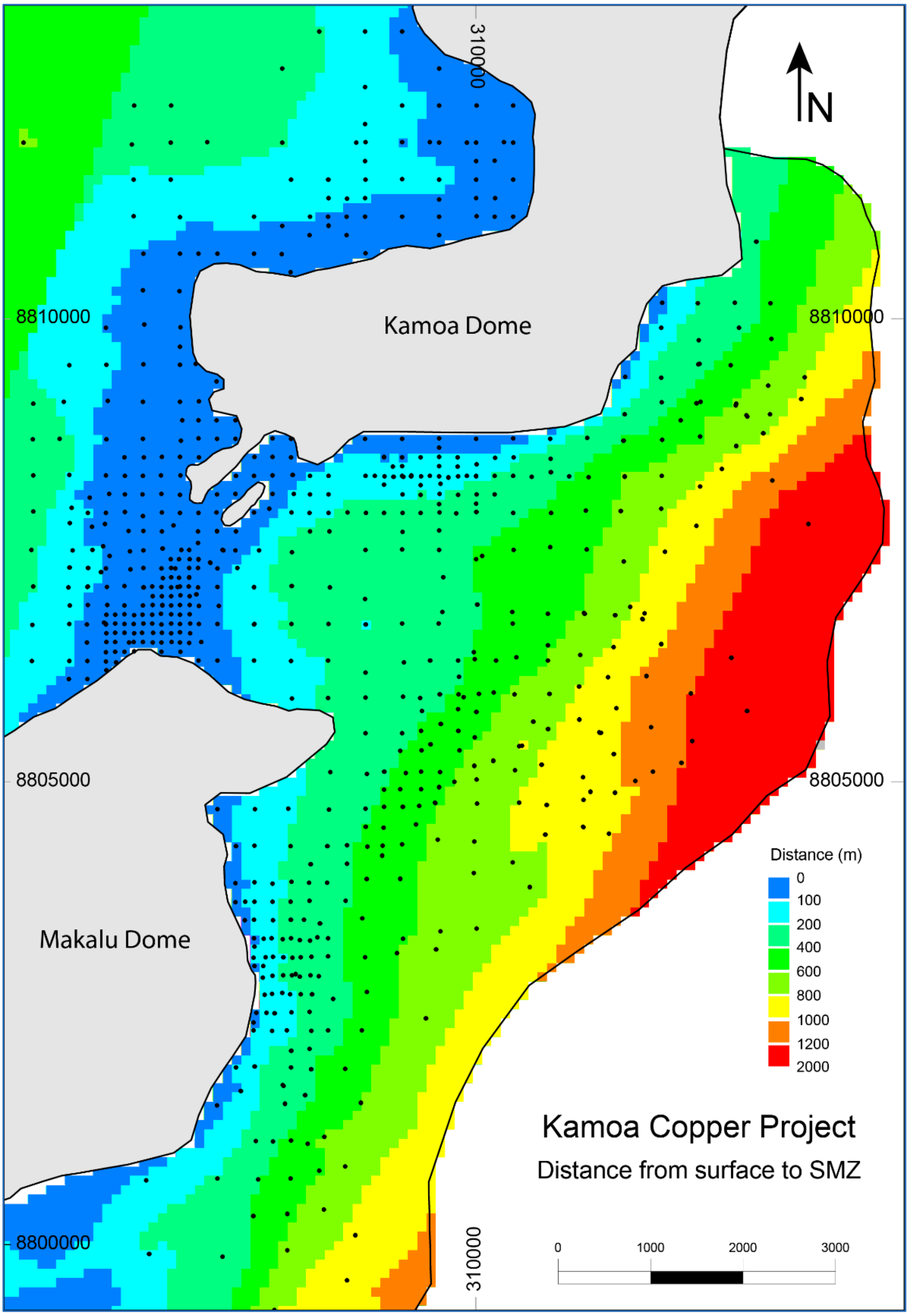
The elevation of the block centroid was set to the elevation of the SMZ DTM at the corresponding easting and northing co-ordinate. A DTM was then created using the centroids of the block model to allow the dip and dip direction of the surface to be estimated from the center-of-gravity point for each triangle in the wireframe. These values were estimated into the block model and tagged to the drillhole composites and used to estimate the true thickness and vertical thickness of each SMZ for each individual drillhole. The block height was set to the estimated vertical thickness, Figure 14.8, and the depth below topography was calculated as shown in Figure 14.9.

Figure 14.8 Schematic Section Showing the Variable Block Height Relative to the Defined SMZ



Note: Figure prepared by Ivanhoe 2014; stratigraphic units in holes: blue = R4.2, dark green = Ki 1.1.1, Yellow = Ki 1.1.2, light green = Ki 1.1.3; copper grade intensity shown by bars on right side of hole. Scale bar represents horizontal metres.

Figure 14.9 Depth of SMZ (Metres below Surface)



Note: Figure prepared by Ivanhoe 2016. Scale bar represents metres.

TCu and ASCu were first weighted by the true thickness (TT) to obtain the variables TCuxTT and ASCuxTT; these variables were estimated into each block using inverse distance weighting to the second power (ID2) interpolation. TCu and ASCu were estimated by dividing the TCuxTT by true thickness and ASCuxTT by true thickness respectively. Estimation parameters are summarized in Table 14.4.

Table 14.4 Estimation Parameters

Pass	Orientation			Search Range	Number of Samples		Estimation Method
	Axis	Azimuth	Dip		Minimum	Maximum	
1	X	90	0	500	4	18	ID2
	Y	0	0	500	4	18	ID2
	Z	0	90	500	4	18	ID2
2	X	90	0	1,250	4	18	ID2
	Y	0	0	1,250	4	18	ID2
	Z	0	90	1,250	4	18	ID2
3	X	90	0	5,000	1	12	ID2
	Y	0	0	5,000	1	12	ID2
	Z	0	90	5,000	1	12	ID2

14.7 Specific Gravity

Specific gravity (SG) was estimated using ID2, using only those SG samples that occurred within the mineralized horizon (refer to discussions in Section 11 on SG determinations). Specific gravity samples were selected within a wireframe constructed on the top and bottom positions of the SMZ blocks. The process was also repeated within the 0.3 m dilution skins (see Section 14.9). A default SG of 2.7 was applied to any block that did not obtain a value from the estimation.

14.8 Resource Models

A model based on intercepts for SMZ10 is the base case model that is used to estimate and report Mineral Resources.

Three sensitivity models were estimated using SMMZ10, SMMZ15, and SMZ20 intercepts (see Table 14.1). These models are contained within the SMZ10 model. The SMMZ15 model was used as the basis for mine planning and Mineral Reserve estimation (refer to Sections 15 and 16).

14.9 Dilution Skins

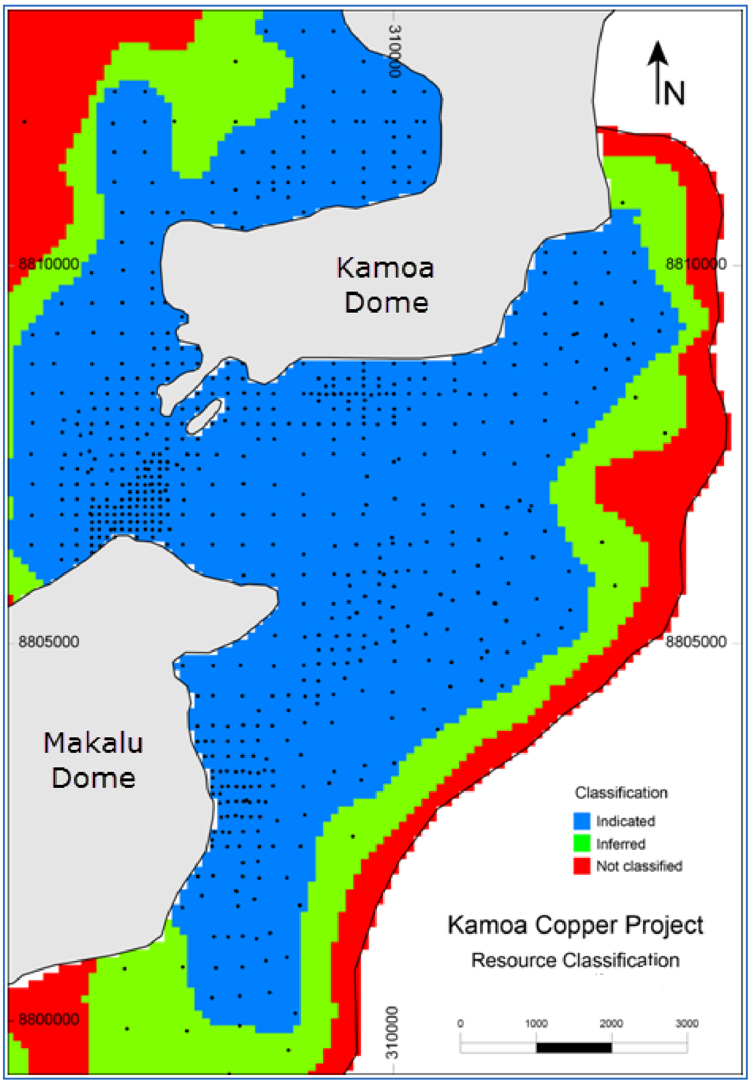
Dilution skins 0.3 m thick were applied to the top (hangingwall) and bottom (footwall) contacts of the SMZ model. Grades were estimated into these blocks using only samples that occurred within the 0.3 m interval. The dilution skins were constructed to allow mining studies to use an estimated dilution grade relevant to the material being added.

14.10 Mineral Resource Classification

Areas outlined by core drilling at 800 m spacing with a maximum extrapolation distance of 600 m between drill sections, and which show continuity of grade at 1% Cu, geological continuity, and continuity of structure (broad anticline with local discontinuities that are likely faults) were classified as Inferred Mineral Resources over an area of 16.8 km². Mineral Resources within an area of 50.5 km² 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 Project is approximately 410.1 km².

Mineral Resources were classified using criteria that required a nominal 400 m drillhole spacing for Indicated and a nominal 800 m drillhole spacing for Inferred. These criteria have been used since 2009. The resource classification is shown in Figure 14.10.

Figure 14.10 Mineral Resource Classification for the Kamoa Project



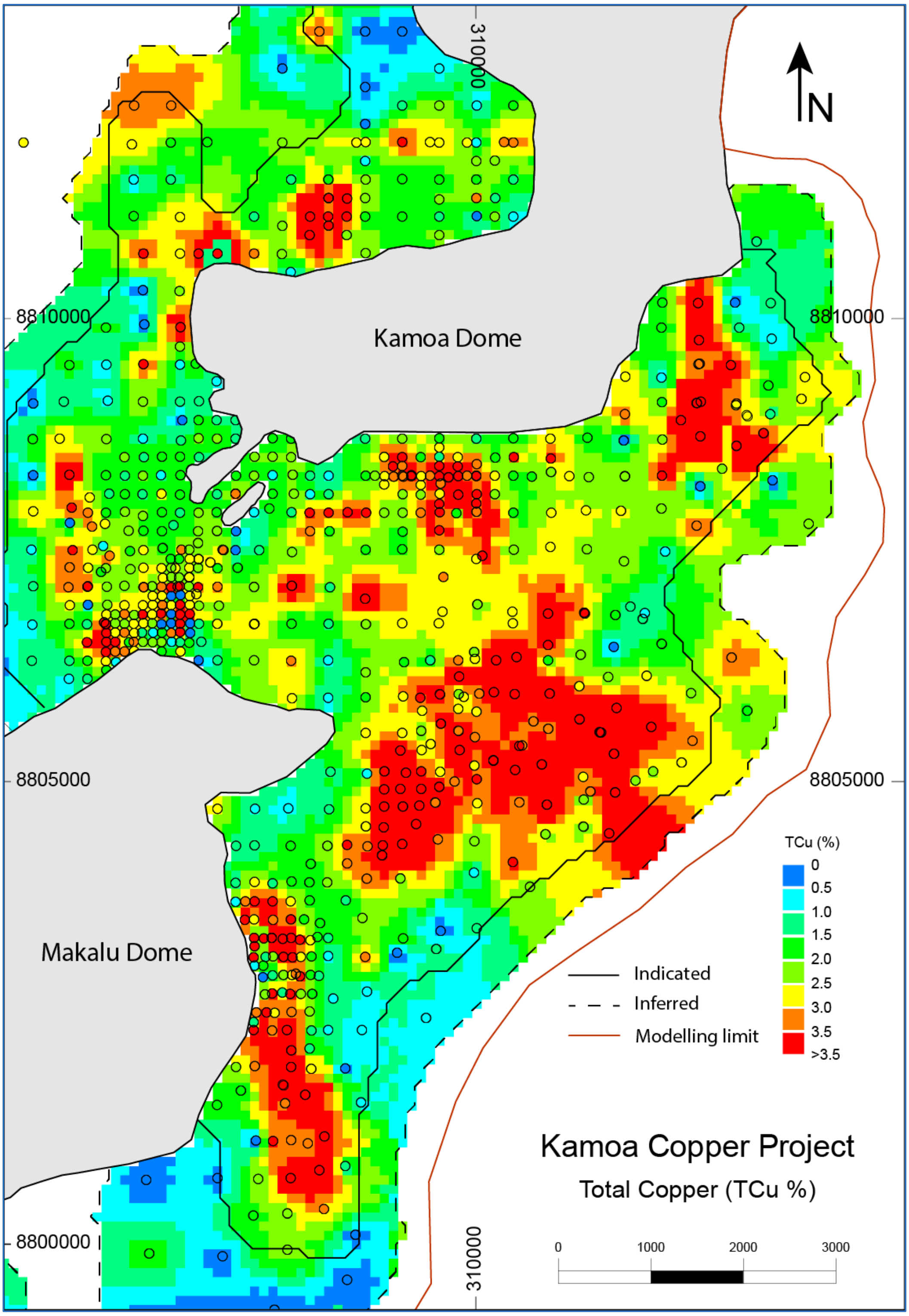
Note: Figure prepared by Ivanhoe 2016. Scale bar represents metres.

14.11 Model Validations

14.11.1 Visual Checks

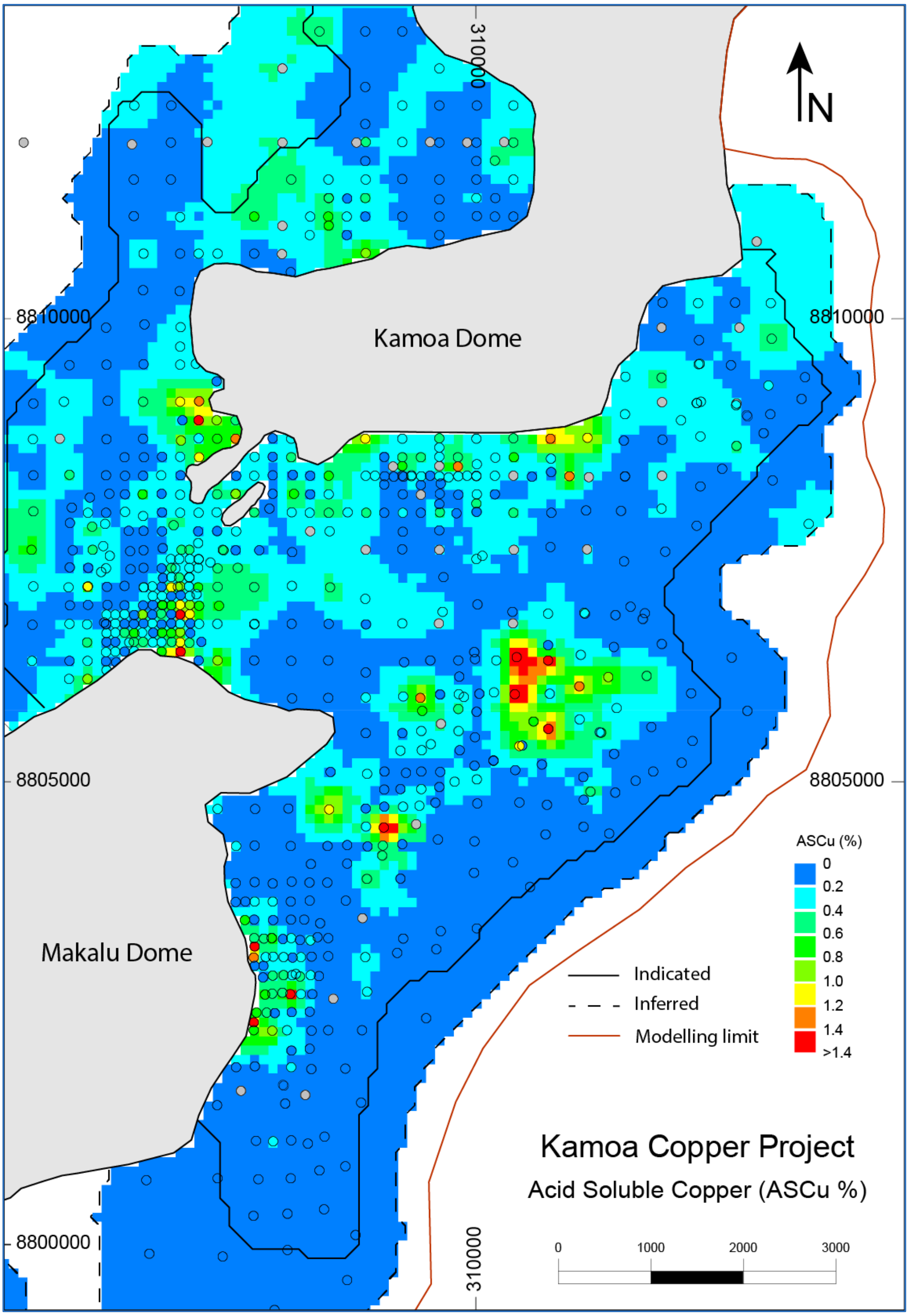
Estimated block grades and composite grades were compared visually in plan view and showed a good agreement, refer to Figure 14.11 through Figure 14.15. The updated model was also visually compared with the 2013 model. The two models agreed very closely with each other in areas where no new drillhole composites had been included. In areas where new drilling has been included, the observed changes in grade were consistent with the new composite grades.

Figure 14.11 Estimated TCu Grade (%) for a 1.0% TCu Modelling Cut-off (SMZ10)



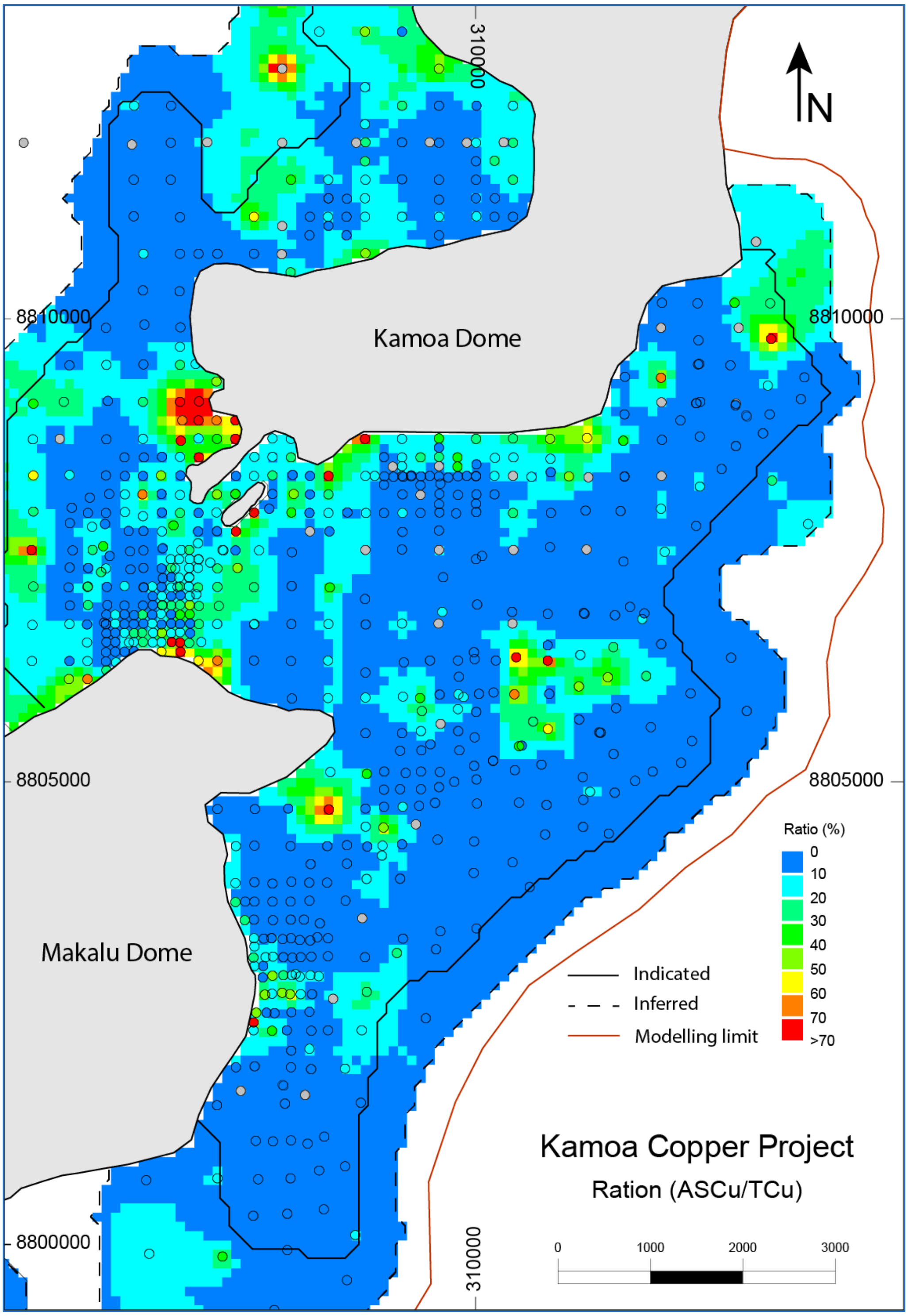
Note: Figure prepared by Ivanhoe 2016. Scale bar represents metres.

Figure 14.12 Estimated ASCu Grade (%) for a 1.0% TCu Modelling Cut-off (SMZ10)



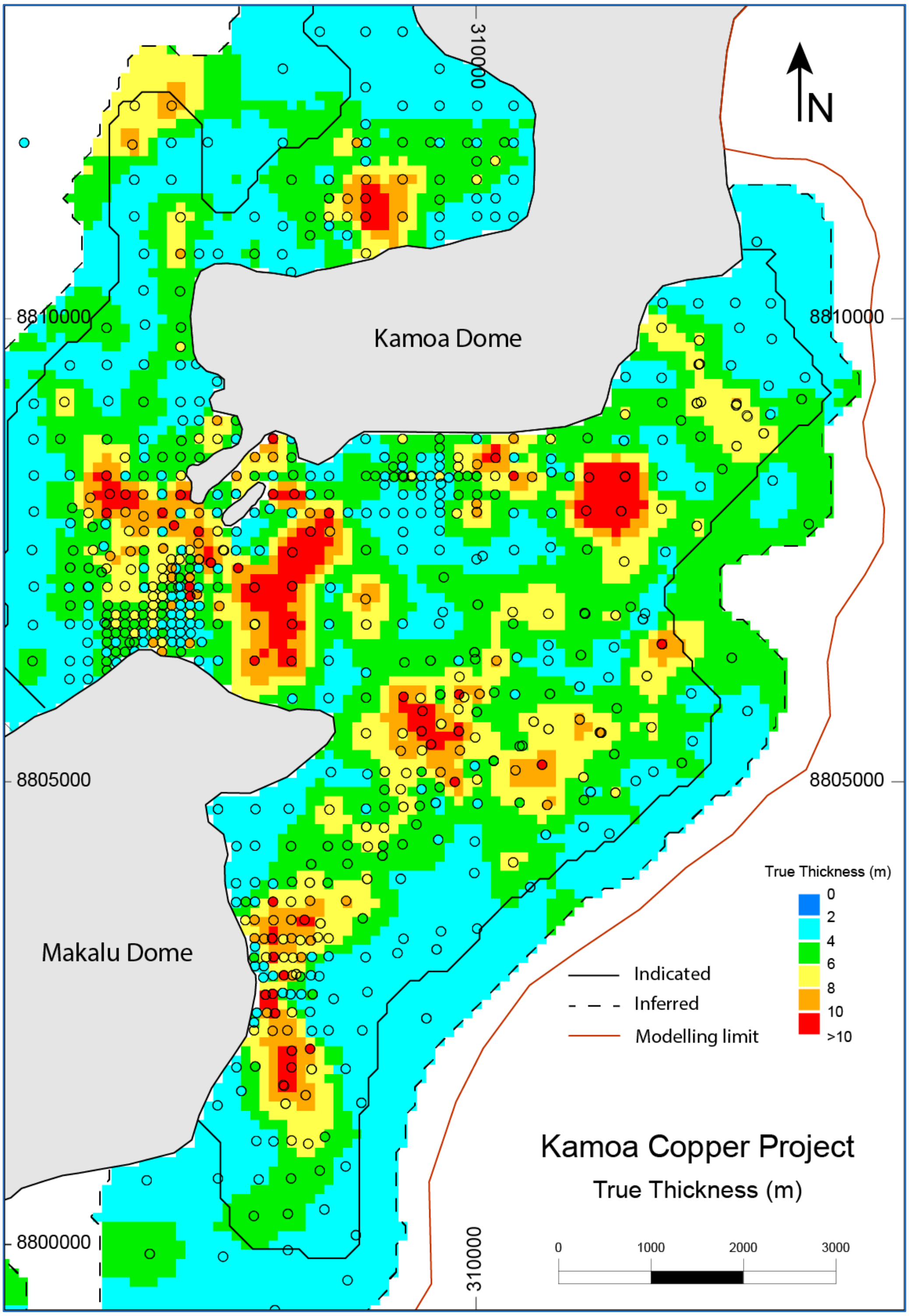
Note: Figure prepared by Ivanhoe 2016. Scale bar represents metres.

Figure 14.13 Estimated Ratio (ASCu:TCu) for a 1.0% TCu Modelling Cut-off (SMZ10)



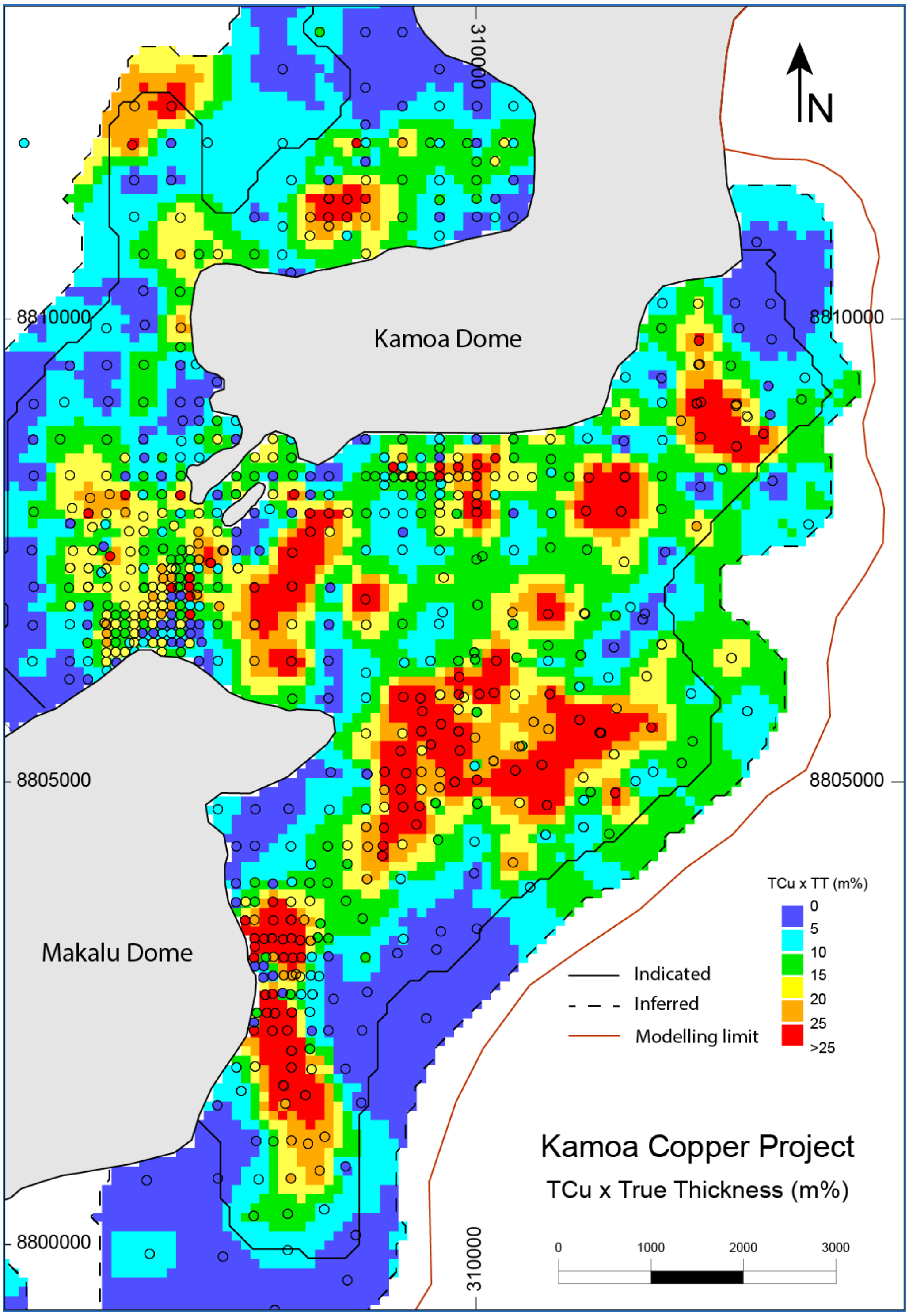
Note: Figure prepared by Ivanhoe 2016; note that the ratio does not appear to be related to depth. Locally the ratio exceeds 50%. Scale bar represents metres.

Figure 14.14 Estimated True Thickness (m) for a 1.0% TCu Modelling Cut-off (SMZ10)



Note: Figure prepared by Ivanhoe 2016. Scale bar represents metres.

Figure 14.15 Estimated TCu Multiplied by True Thickness (TCuxTT, m%) for a 1.0% TCU Modelling Cut-off (SMZ10)



Note: Figure prepared by Ivanhoe 2016.

14.12 Global Bias Checks

14.12.1 Global Bias

During estimation, a nearest neighbour (NN) estimate was included to allow a check for global bias between the estimated grade and the drillhole grades. The NN estimate minimizes the effect of clustering of data and allows for a more appropriate comparison with estimated grades. This is very 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 ID2 and NN models are below 5%, which is considered appropriate for an Indicated classification, Table 14.5.

Table 14.5 Mean Grades for 1.0% Cut-off (SMZ10) Composites and Models

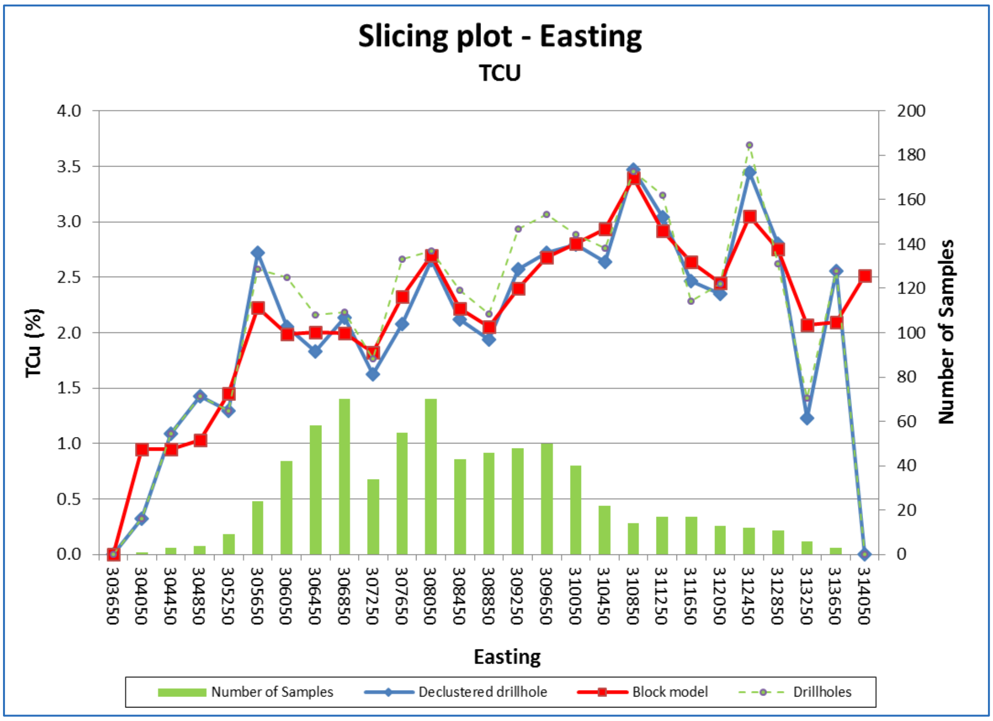
Indicated (no cut-off applied)	Composite	Model (ID2)	Model (NN)	Relative Diff (ID2-NN)/NN
TCu x TT	15.07	13.17	12.96	1.6%
ASCU x TT	1.68	1.31	1.30	0.6%
True Thickness	5.57	5.10	5.09	0.4%
Inferred				
TCu x TT	3.82	6.09	5.51	10.6%
ASCU x TT	0.53	0.64	0.62	2.5%
True Thickness	3.39	3.66	3.57	2.4%

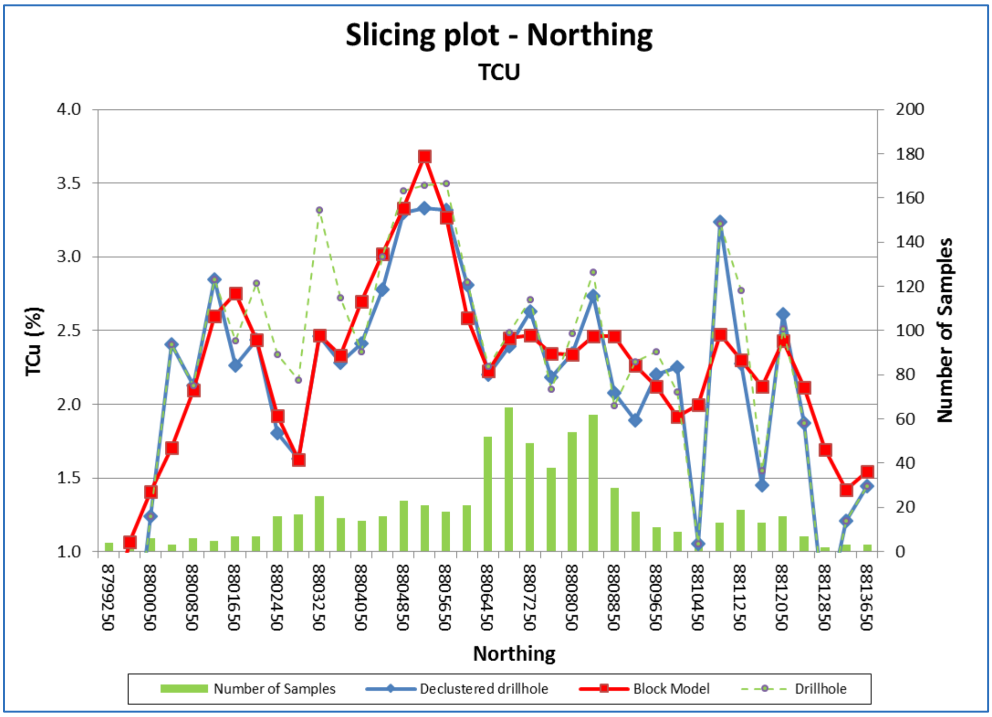
Note: Composite values are higher for ASCUxTT due to clustering of data in the shallow mineralization between the Kamoa and Makalu Domes.

14.12.2 Local Bias Checks (Swath Plots)

Checks for local bias were performed for TCu, ASCu and true thickness by analyzing local grade trends on 400 m slices (swaths) in easting and northing. Example swath plots for TCu (%) are shown in Figure 14.16. The average grade per swath for the block model (red line) is compared with the average grade for the declustered drillholes (blue line) for the same swath. The two lines are observed to follow very similar trends, indicating that no local biases are evident.

Figure 14.16 Swath Plots for TCu (%) for the 1.0% TCu Cut-off Model (SMZ10)





Note: Figures prepared by Ivanhoe 2014.

14.13 Reasonable Prospects of Eventual Economic Extraction

Amec Foster Wheeler has used a 1% TCu cut-off grade to support Mineral Resource estimation. This choice of cut-off is based on many years of mining experience on the Zambian Copperbelt at mines such as Konkola, Nchanga, Nkana, and Mufulira, which mine similar mineralisation to that identified at Kamoa.

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$3.30/lb was assumed, which was the copper price used by Peters et al. (2013) at the time the resource model was constructed, and in the opinion of the QPs is still valid. The conceptual evaluation did not include provisions for mining or processing credits, such as acid credits.

The following additional key parameters were used:

- Percent Recovery for Supergene = 86.01(1-exp(-0.9173TCu)), based on 2013 PEA milestone flowsheet.
- Percent Recovery for Hypogene = -0.00725x6 + 0.2066x5 -2.34998x4 + 13.63868x3-42.90156x2 +71.82531x +33.49319, where x = TCu; based on IS4A hypogene flowsheet.
- Concentrate grades for Supergene 45.1% TCu and 14.3% S.
- Concentrate grades for Hypogene 34.7% TCu and 26.7% S.
- Concentrate moisture of 12%.
- Mining costs of US$34/t.
- Concentrator, tailings and G&A costs of US$19/t treated.
- Payable copper of 96.7%.
- Smelting costs of US$80/t of concentrates.
- Refining costs of 0.08$/lb payable copper.
- NSR = Payable TCu – Smelting Costs – Refining Costs
- Transport costs of concentrates to smelter $300/wmt concentrates.
- Royalty of 2% on NSR – Transport.
- National Export Tax of 1% of NSR.
- Provincial Export tax of $60/wmt concentrates.

The supergene recovery equation is derived from testwork based on samples taken throughout the area of the deposit where Mineral Resources have been estimated. The hypogene recovery equation is based on Figure 13.14; it confirms 2013 testwork and is assumed to be applicable to the hypogene mineralization throughout the deposit.

The remaining assumptions follow those in Section 22.

Normally, cut-off grades used to declare Mineral Resources do not consider mining costs; however, in that case, Mineral Resources are declared within stope blocks, which have only been defined (see Section 15) above a nominal 1.5% TCu cut-off and a $100/t NSR in the Kamoa 2016 PFS. There are additional areas for which reasonable prospects for eventual economic extraction exist and which might be scheduled if the nominal 3 Mtpa production rate used for the Kamoa 2016 PFS was increased to as much as 20 Mtpa. These additional areas are included using a 1% TCu cut-off. There is a small percentage (~10%) of the tonnage that has copper grades between 1.0% and 1.5%; these blocks will not cover their full 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). A savings would be realized 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 $250/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 percentage of blocks that will not cover full mining costs decreases to less than 5%. Again it may be convenient to mine these blocks in conjunction with adjacent higher-grade blocks.

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 fill is utilized. In addition, the Mineral Resources do not incorporate allowances for contact (external) dilution at the roof and floor of the deposit. This has been estimated to be approximately 30 cm at the hangingwall and footwall contacts (see Section 16.2.2), but will ultimately depend on the ability of the mining to follow the SMZ boundaries.

14.14 Mineral Resource Statement

The Mineral Resources were classified in accordance with the 2014 CIM Definition Standards. Mineral Resources are stated in terms of total copper (TCu), and an approximate minimum true thickness of 3 m. Indicated and Inferred Mineral Resources for the SMZ10 resource model are summarized in Table 14.6. Mineral Resources are inclusive of Mineral Reserves.

The Mineral Resources have an effective date of 5 May 2014. The Mineral Resources do not include any material in the hangingwall and footwall dilution skins and make no allowance for mining recovery factors.

Table 14.6 SMZ10 Kamoa Project Indicated and Inferred Mineral Resource (at 1% TCu Cut-off Grade)

Category	Tonnage (Mt)	Area (km^2)	Copper (%)	True Thickness (m)	Contained Copper (kt)	Contained Copper (billions lbs)
Indicated	752	50.5	2.67	5.24	20,110	44.3
Inferred	185	16.8	2.08	3.87	3,840	8.5

1. Dr. Harry Parker and Gordon Seibel, both RM of SME, employees of Amec Foster Wheeler, are the Qualified Persons for the Mineral Resource estimate. The effective date of the estimate is 5 May 2014. Mineral Resources are reported inclusive of Mineral Reserves.
2. Mineral Resources are reported using a total copper (TCu) cut-off grade of 1% TCu and an approximate minimum true thickness of 3 metres. There are reasonable prospects for eventual economic extraction under assumptions of a copper price of US$3.30/lb; employment of underground mechanized 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 $34/t. Concentrator and General and Administrative costs are assumed to be $19/t. Metallurgical recovery will be 77% (supergene) and 85% (hypogene) at the average grade of the resource.
3. Reported Mineral Resources contain no allowances for hangingwall or footwall contact boundary loss and dilution. No mining recovery has been applied.
4. For Indicated Mineral Resources, 97.4% of the resource model blocks have a true thickness greater than 3 metres (range from 2.3 m to 15.8 m), for Inferred Mineral Resources, 94.7% of the resource blocks have a true thickness greater than 3 m (range from 2.7 m to 8.4 m).
5. Depth of mineralization below the surface ranges from 10 metres to 1,320 metres for Indicated Mineral Resources and 20 metres to 1,560 metres for Inferred Mineral Resources.
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.

14.15 Sensitivity of Mineral Resources to Cut-off Grade

Table 14.7 summarizes the Mineral Resource for the 1.0% TCu cut-off model (SMZ10). The base case Mineral Resource model reported at a 1.0% TCu cut-off is highlighted in grey.

Table 14.7 Sensitivity of Mineral Resources to Cut-off Grade

Indicated Mineral Resource					
Cut-off (%Cu)	Tonnage (Mt)	Area (km²)	Copper (%)	Contained Copper (kt)	Contained Copper (billions lbs)
4.00	77	4.4	4.61	3,565	7.9
3.50	147	8.5	4.19	6,169	13.6
3.00	238	14.2	3.83	9,120	20.1
2.50	383	23.9	3.42	13,090	28.9
2.00	550	34.4	3.07	16,880	37.2
1.75	628	39.6	2.92	18,330	40.4
1.50	686	43.8	2.81	19,280	42.5
1.25	722	47.3	2.74	19,770	43.6
1.00	752	50.5	2.67	20,110	44.3
0.80	769	52.4	2.64	20,260	44.7
0.60	776	53.2	2.62	20,310	44.8
Inferred Mineral Resource					
Cut-off (%Cu)	Tonnage (Mt)	Area (km²)	Copper (%)	Contained Copper (kt)	Contained Copper (billions lbs)
4.00	2	0.2	4.47	89	0.2
3.50	7	0.7	3.94	287	0.6
3.00	20	1.7	3.45	700	1.6
2.50	52	4.1	3.00	1,570	3.5
2.00	93	7.6	2.67	2,470	5.4
1.75	117	10.0	2.51	2,920	6.4
1.50	136	11.9	2.38	3,240	7.1
1.25	159	14.2	2.23	3,560	7.8
1.00	185	16.8	2.08	3,840	8.5
0.80	209	19.3	1.95	4,060	8.9
0.60	225	21.1	1.86	4,170	9.2

14.16 Considerations for Mine Planning

A review of the block model was undertaken, applying higher cut-offs in defining the SMZs to allow for high-grade, narrower, mining options to be tested in support of potentially improving initial mine economics. An assumption was made that the maximum mining constraint would be 6 m (a single mining cut), and that the highest grade material would be extracted by subsequent cuts. Figure 14.17 illustrates the nature of the grade profile as the mineralisation is generally higher-grade towards the bottom. In some cases, particularly in the southern Makalu areas where the Ki1.1.1 thickens significantly, the highest-grade interval shifts to what is obviously a different stratigraphic position.

Figure 14.17 Impact of the Adjustments in SMZ Coding for Different Grade Profiles

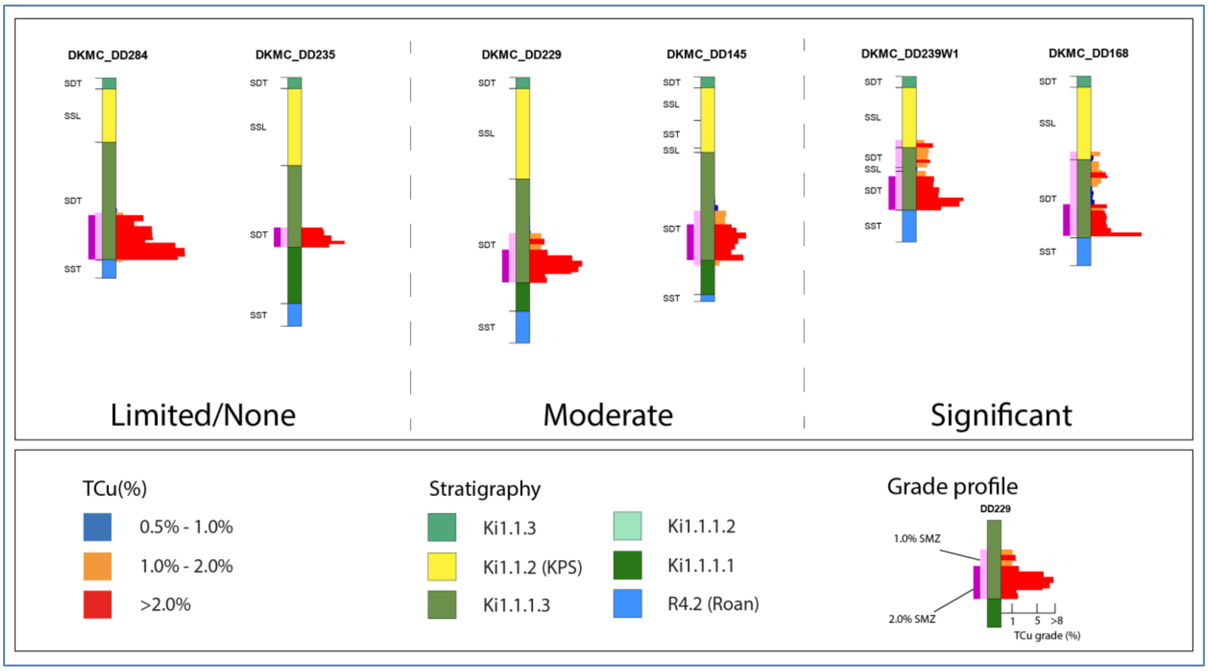


Figure prepared by Ivanhoe 2014

As expected, the mean TCu grade increases at higher cut-offs, with a corresponding decrease in true thickness. The coefficient of variation (CV), increases gradually at higher cut-offs. At higher cut-offs, where the SMZ narrows, the continuity of the higher-grade zones actually improves. Figure 14.18 through Figure 14.21 show model TCu grades for selected cut-off grades and illustrate the relative continuity of grades above those cut-offs. Following a review of the various cut-offs, it was decided to proceed with mine planning using a 1.5% TCu cut-off grade. The minimum thickness was 3.0 m. The maximum thickness was 6.0 m; this was relaxed if material above 2.0% TCu was encountered above the 6.0 m limit. Figure 14.20 shows this case.

Figure 14.18 Grades for the 1.0% TCu cut-off model (SMZ criteria = >1% TCu and >3 m thickness)

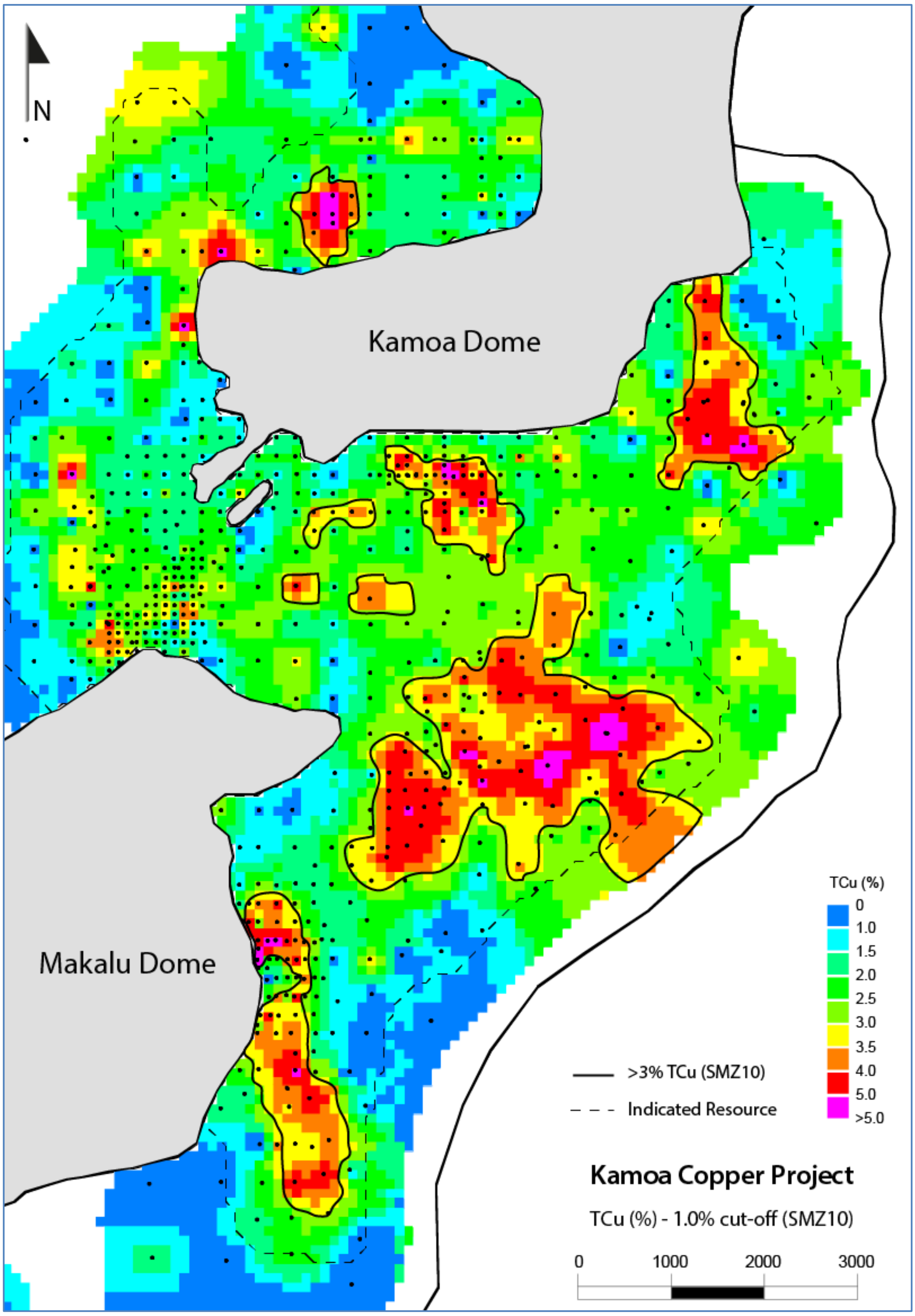


Figure prepared by Ivanhoe 2014. Scale bar represents metres.

Figure 14.19 Grades for the 1.0% TCu cut-off model (SMZ criteria = >1% TCu and >3 m and ≤6 m thickness unless > 2.0% TCu above 6 m)

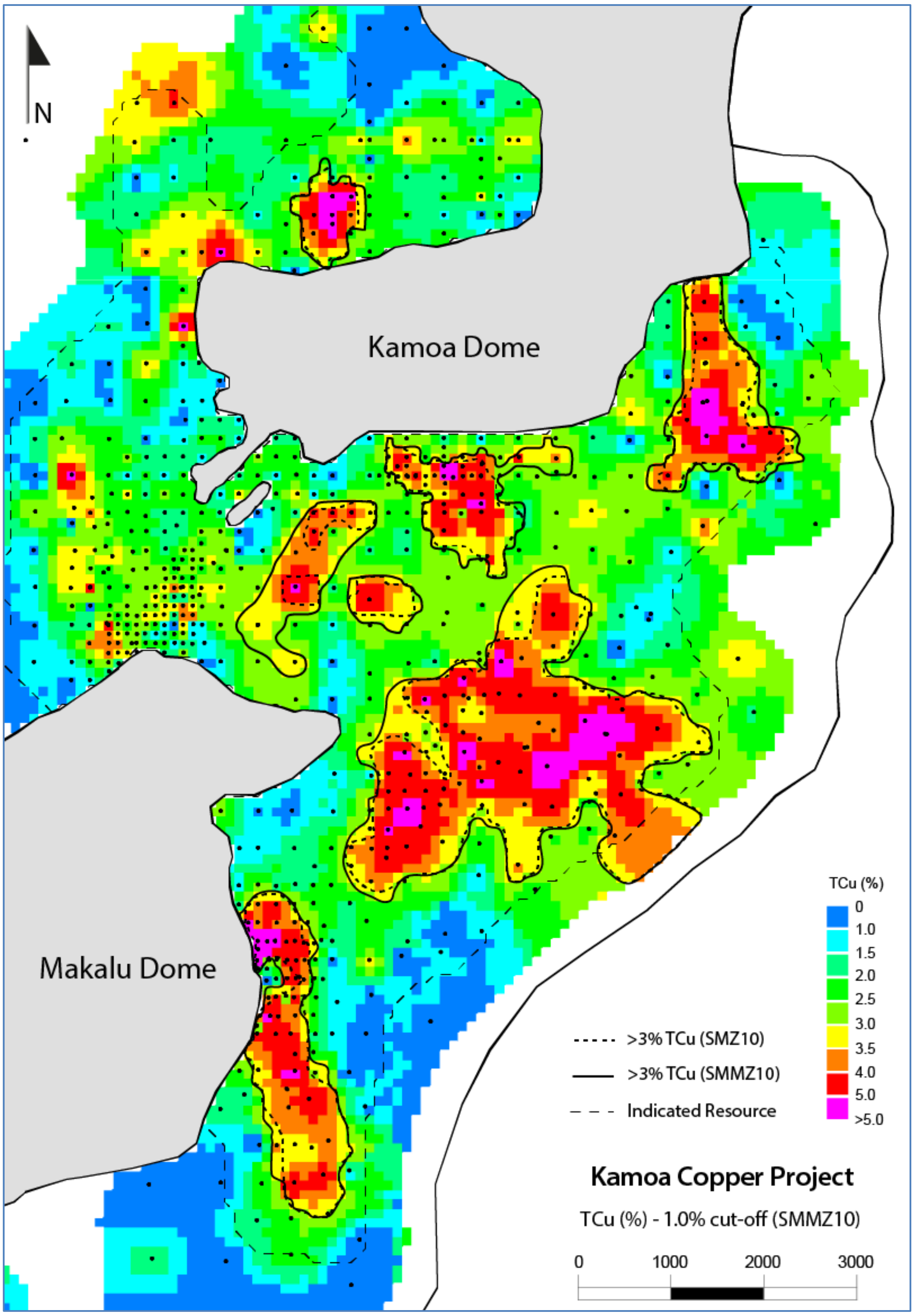


Figure prepared by Ivanhoe 2014; SMZs limited to 6 m unless material above grades >2% Cu. Scale bar represents metres. SMZ10 represents >1.0% TCu over 3 m; SMMZ10 represents 1.0% TCu over 3.0 m limited to 6.0 m unless material grades above >2% Cu.

Figure 14.20 Grades for the 1.5% TCu cut-off model (SMZ criteria = >1.5% TCu and >3 m and ≤6 m thickness unless > 2.0% TCu above 6 m)

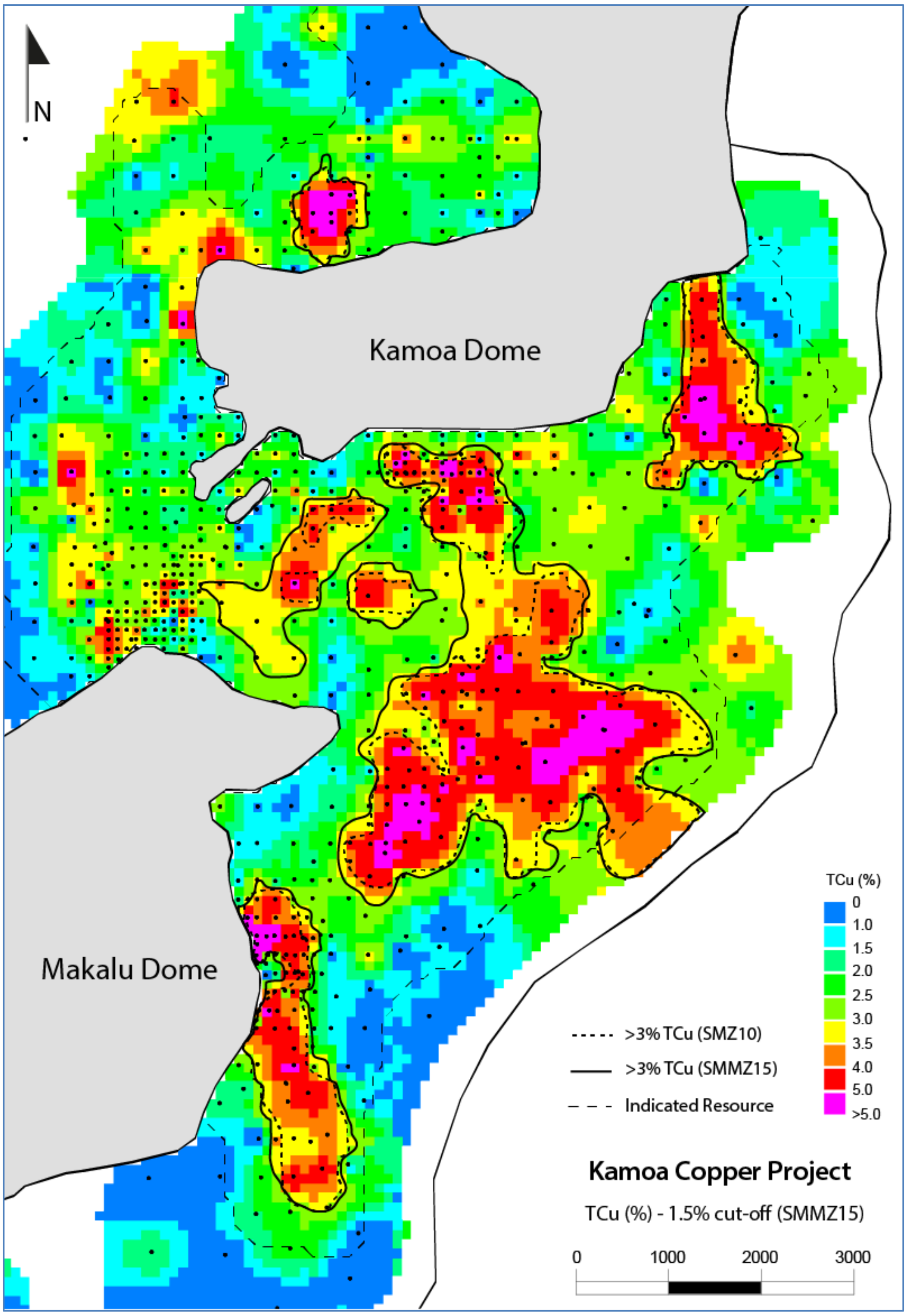


Figure prepared by Ivanhoe 2014; SMZs limited to 6 m unless material above grades >2% Cu. Scale bar represents metres. SMZ10 represents >1.0% TCu over 3 m; SMMZ15 represents 1.5% TCu over 3.0 m limited to 6.0 m unless material grades above >2% Cu.

Figure 14.21 Grades for the 2.0% TCu cut-off model (SMZ criteria = >2% TCu and >3 m thickness)

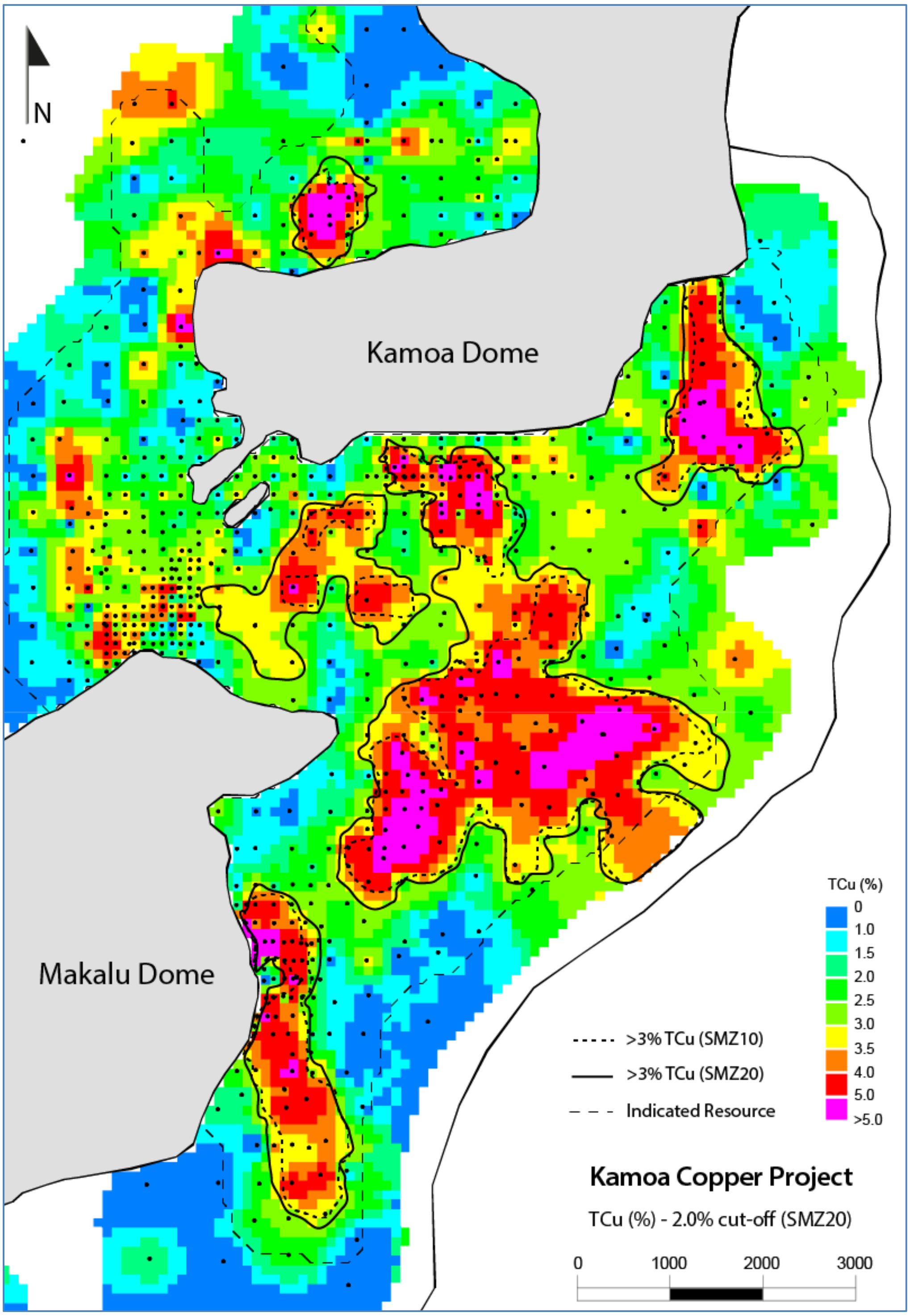


Figure prepared by Ivanhoe 2014; SMZs limited to 6 m unless material above grades >2% Cu. Scale bar represents metres. SMZ10 represents >1.0% TCu over 3 m; SMMZ20 represents 2.0% TCu over 3.0 m.

14.17 Targets for Further Exploration

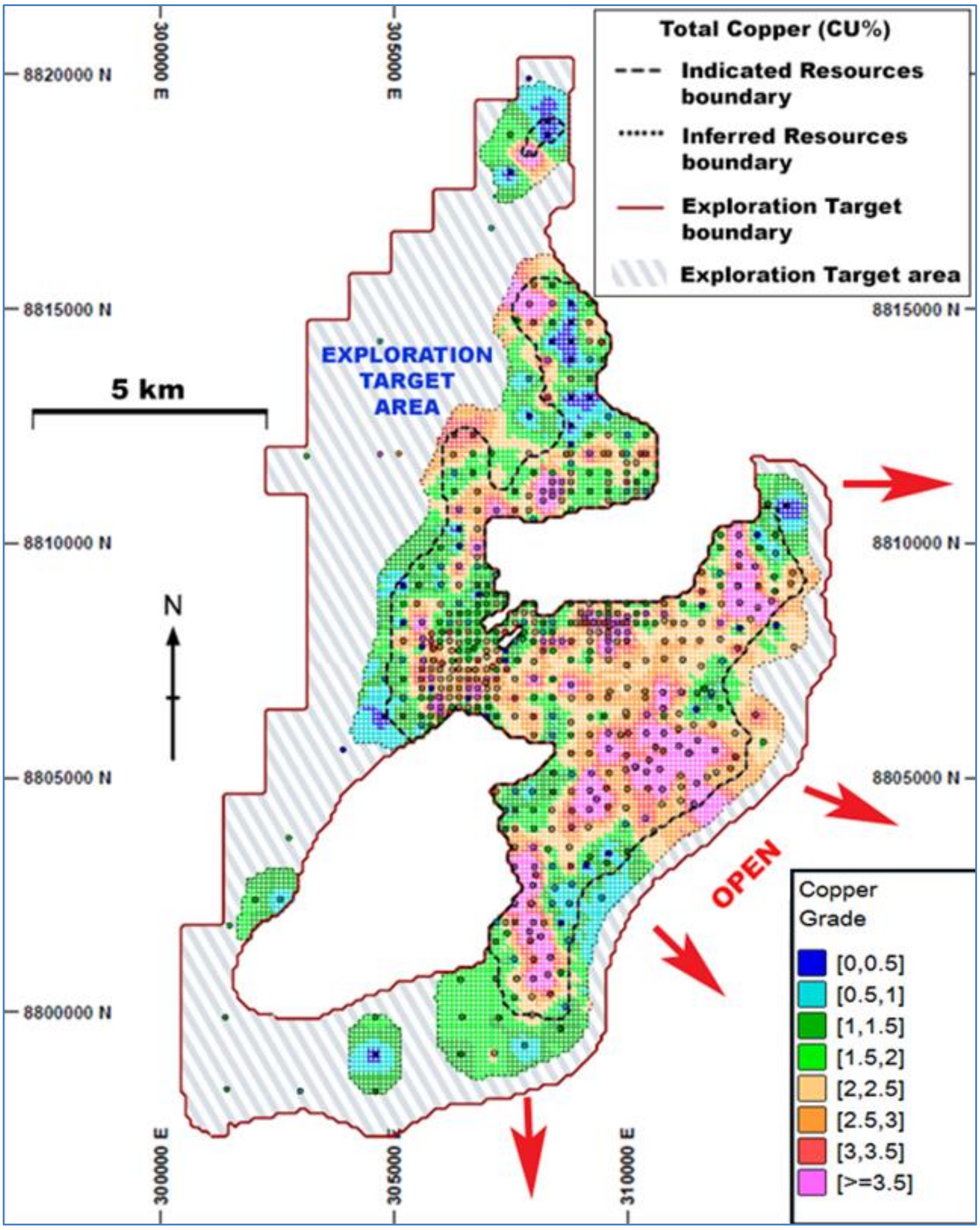
Amec Foster Wheeler has identified two targets for further exploration (referred to as exploration targets for the purposes of this Report). They are referred to in this subsection as the Kamoa–Makalu exploration target, and the Kakula Discovery exploration target.

14.17.1 Kamoa–Makalu

The area inside the model perimeter surrounding the Indicated and Inferred Mineral Resources is shown in Figure 14.22. The ranges of exploration target tonnages and grades are summarised in Table 14.8. 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 14.22 Location of Kamoa–Makalu Target for Further Exploration and Drillholes



Note: Figure by Ivanhoe, 2016. Scale bar represents metres.

Table 14.8 Tonnage and Grade Ranges for Kamoa Makalu Target for further Exploration

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

14.17.2 Kakula Exploration Area

Figure 14.23 shows the location of the Kakula Exploration area is located about 5 km southwest of Kamoa. Following up a geochemical anomaly, Ivanhoe has found thick mineralization within eight holes along a north–west-trending corridor (called the Discovery Target area), as shown in Figure 14.24. The mineralization occurs along the contact between the Roan (4.2) and a massive diamictite (Ki 1.1.1), with upward zoning of chalcocite, bornite and chalcopyrite. Figure 14.25 shows two cross sections, and Figure 14.26 provides a grade profile for drillhole DKMC_DD997. This drillhole and its neighbour DKMC-DD996 (highlighted in yellow in Figure 14.23) have among the highest grade-thicknesses encountered to date within the Kamoa project area.

The bottom-loaded nature of the mineralization within the Kakula Discovery exploration target area could support the definition of selective mineralized zones at a 1% and a 2% cut-off. Table 14.9 shows composites using 1% and 2% copper cut-offs using a minimum mining width of 3 metres.

Figure 14.23 Kamoa Project Map showing Kakula Exploration Area

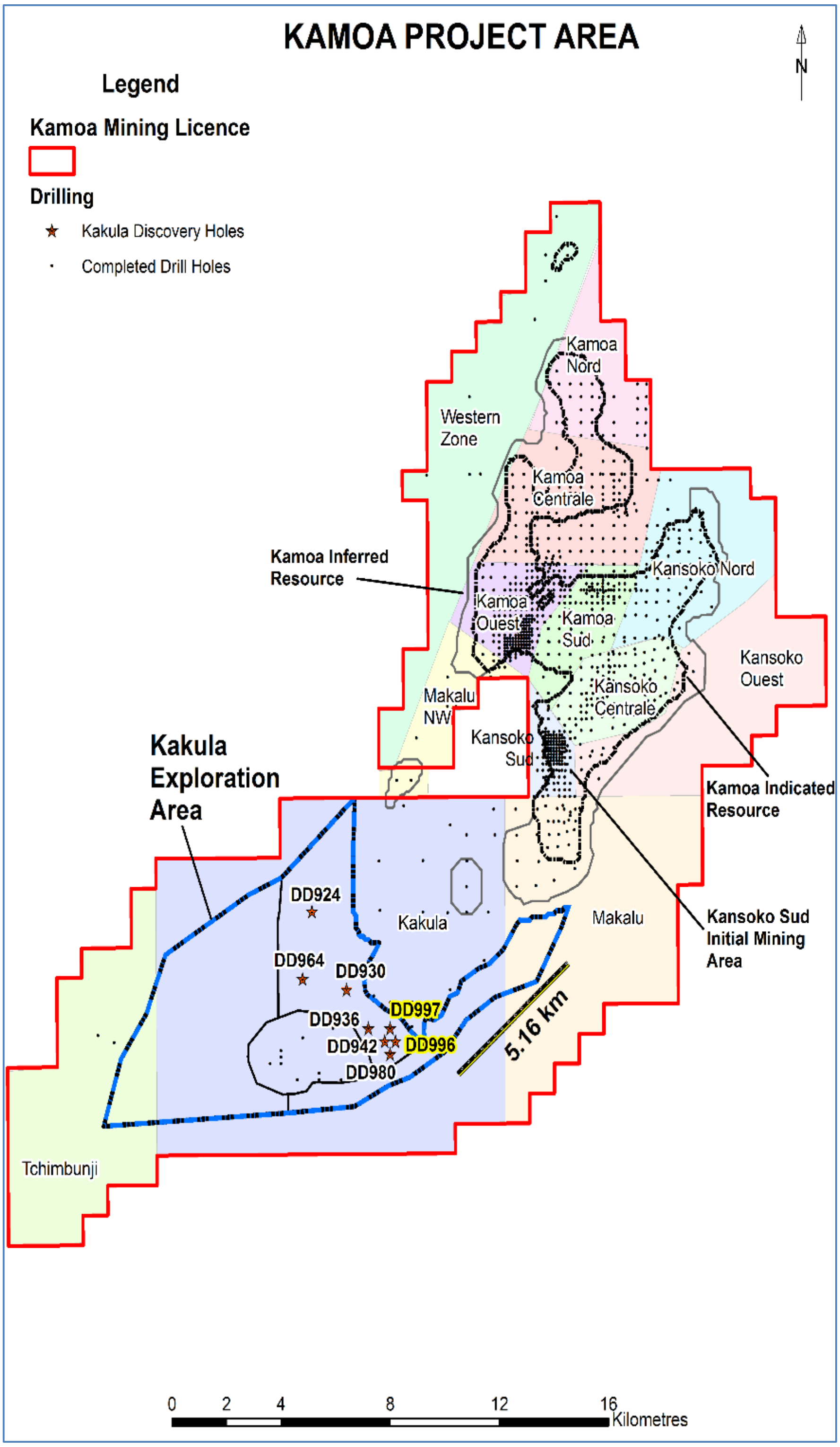


Figure provided by Ivanhoe, 2016; highest grade holes at Kakula highlighted in yellow.

Figure 14.24 Kakula Discovery Target and Areas for Further Exploration

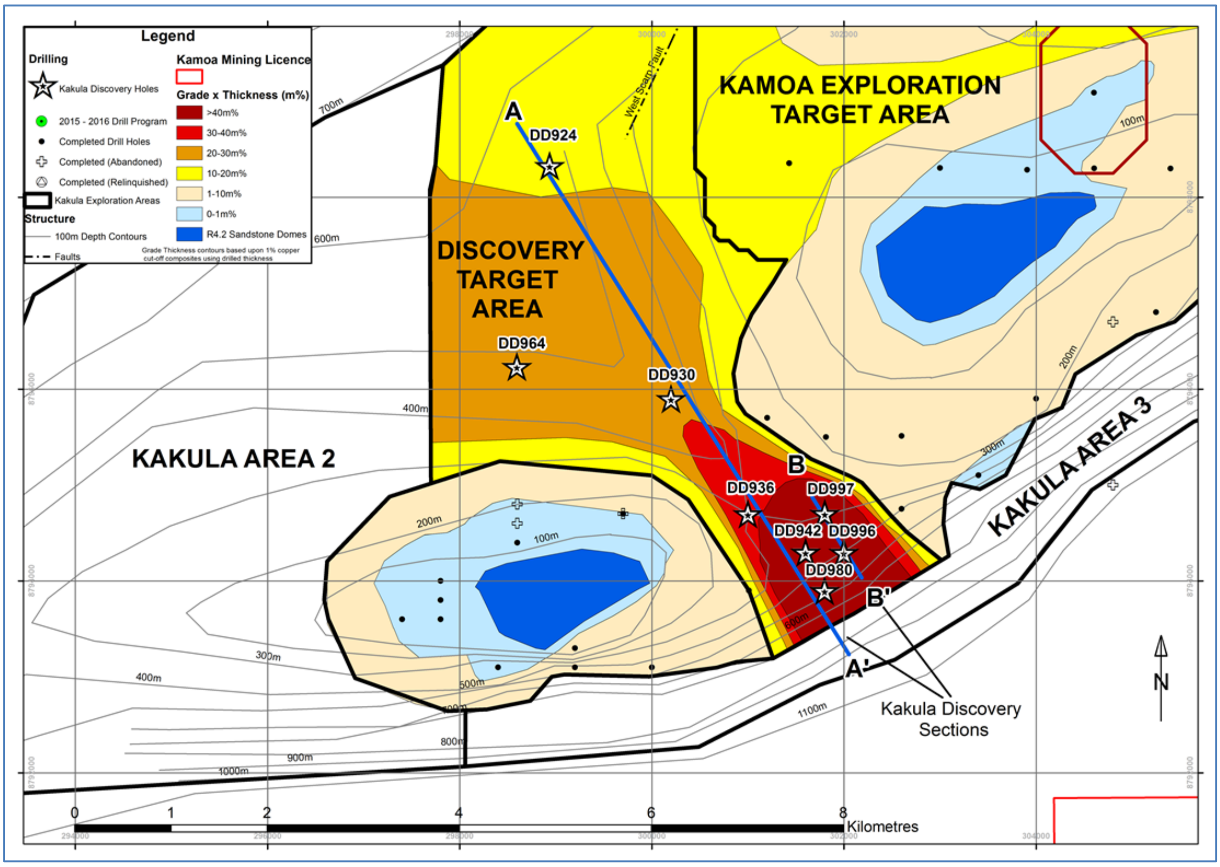


Figure provided by Ivanhoe, 2016. In this figure, the Kamoa–Makalu target for further exploration is labelled as the Kamoa exploration target area. The areas labelled as Kakula Area 2 and Kakula Area 3 have exploration potential.

Figure 14.25 Sections through Kakula Discovery Target (intercepts at 2% Cu Cut-Off)

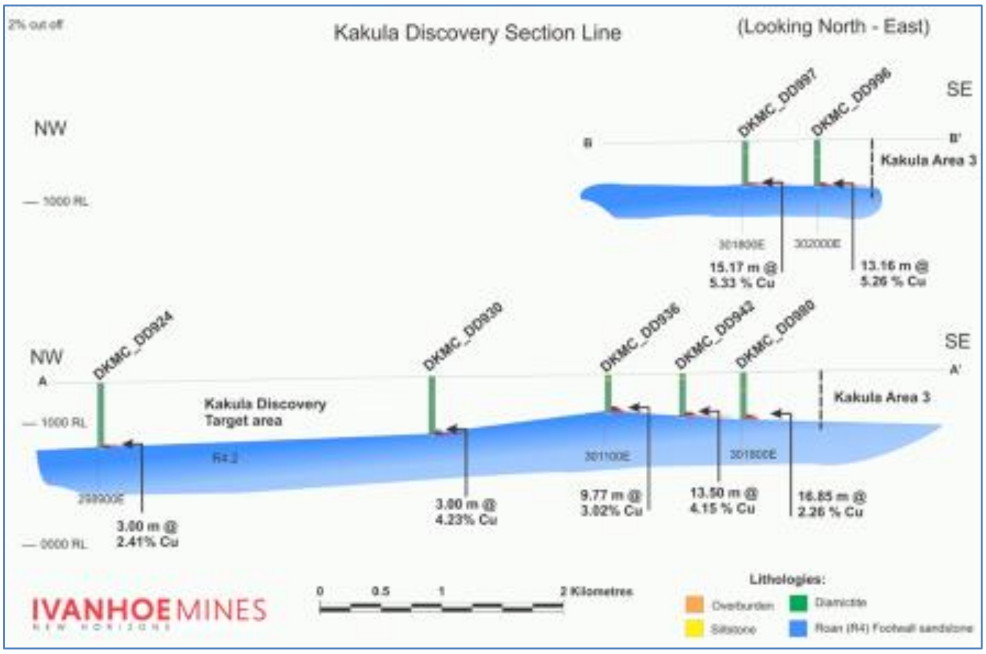


Figure provided by Ivanhoe, 2016.

Figure 14.26 DKMC_DD997 Strip Log showing Bottom loaded Distribution of Copper Mineralisation

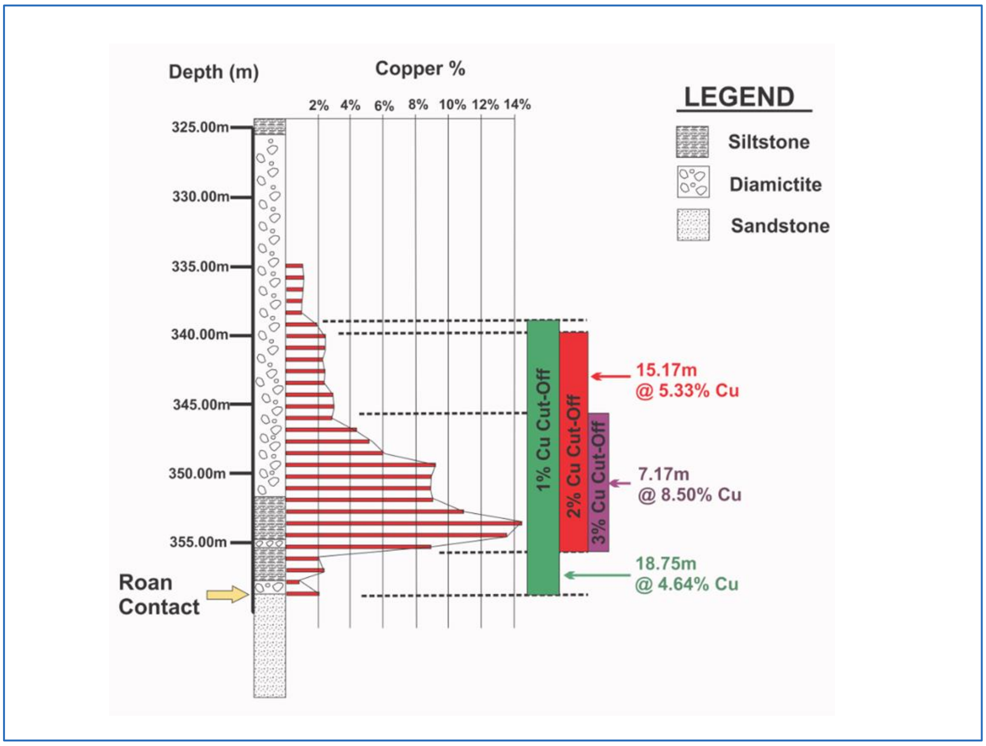


Figure provided by Ivanhoe, 2016.

Table 14.9 Kakula Discovery Target Composites at a 1% and 2% Copper Cut-Off

Hole ID	1% Cut-off				
	From	To	Length (m)	True Width (m)	Grade (Cu %)
DKMC_DD924	513.94	528.50	14.56	14.06	1.68
DKMC_DD930	458.00	472.50	14.50	14.49	2.02
DKMC_DD936	284.45	299.18	14.73	14.73	2.58
DKMC_DD942	317.00	343.30	26.30	26.28	2.88
DKMC_DD964	864.20	883.00	18.80	18.78	1.38
DKMC_DD980	349.16	380.85	31.69	31.64	1.86
DKMC_DD996	353.00	377.16	24.16	24.13	3.48
DKMC_DD997	340.00	358.75	18.75	18.47	4.64
2% Cut-off					
DKMC_DD924	525.00	528.00	3.00	2.90	2.41
DKMC_DD930	469.00	472.00	3.00	3.00	4.23
DKMC_DD936	289.00	298.77	9.77	9.77	3.02
DKMC_DD942	328.00	341.50	13.50	13.49	4.15
DKMC_DD964	878.75	882.00	3.25	3.25	2.17
DKMC_DD980	364.00	380.85	16.85	16.82	2.26
DKMC_DD996	364.00	377.16	13.16	13.14	5.26
DKMC_DD997	341.00	356.17	15.17	14.94	5.33
	3% Cut-off				
DKMC_DD924	No intervals meet cut-off				
DKMC_DD930	470.00	472.00	2.00	2.00	5.53
DKMC_DD936	293.00	297.62	4.62	4.62	3.79
DKMC_DD942	337.80	341.50	3.70	3.70	9.26
DKMC_DD964	No intervals meet cut-off				
DKMC_DD980	No intervals meet cut-off				
DKMC_DD996	371.00	376.6	5.60	5.59	9.16
DKMC_DD997	349.00	356.17	7.17	7.06	8.50

Table provided by Ivanhoe, 2016, modified by Amec Foster Wheeler

The Discovery Target has an areal extent of 19 km^2. Experience at Kamoa and Konkola (in Zambia) has shown that Copperbelt deposits related to carbonaceous shales and siltstones overlying oxidized arkosic sandstones are continuous, and therefore ranges and tonnage and grades can be estimated from a limited number of drillholes. The drilling results shown in Table 14.9 were used, and a success-rate of 80% was applied to reflect uncertainty as to the position of thin or leached mineralization bounding the target. The range of tonnages and grades was then derived by applying a variance factor of +/-20%. Table 14.10 provides ranges of potential tonnage and grades. Amec Foster Wheeler cautions that the potential quantity and grade of the Discovery target for futher exploration is conceptual in nature, and that it is uncertain if additional drilling will result in the exploration target being delineated as a Mineral Resource.

Table 14.10 Tonnage and Grade Ranges for Discovery Target further Exploration

Target	Low-range Tonnage Mt	High-range Tonnage Mt	Low-range Grade (% Cu)	High-range Grade (% Cu)
Total	580	870	1.5	2.3

The Kakula Exploration area contains additional prospective ground, termed Kakula Area 2 and Kakula Area 3 (see Figure 14.24).

The planned Kakula exploration drilling program consists of 19,000 metres in 58 holes, of which approximately 10,000 metres of drilling is planned on an 800-metre infill grid at the Kakula Discovery area so that an Inferred Mineral Resource can be estimated. The other approximately 9,000 metres of drilling will be used to test the other two areas of exploration potential within the Kakula exploration area.

Comment

The Discovery Target is likely a mosaic piece located between two domes (Kakula and Kakula Northeast), which may have constrained the mineralizing fluids. The shape of the mosaic piece reflected by the five thick and high-grade holes (DKMC_DD936, DKMC_DD942, DKMC_DD980, DKMC_DD996, DKMC_997) is an analogue to the Kansoko Sud initial mining area at Kamoa (refer to Figure 14.27).

Figure 14.27 Comparison of Kansoko Sud (Kamoa) and Discovery Target (Kakula)

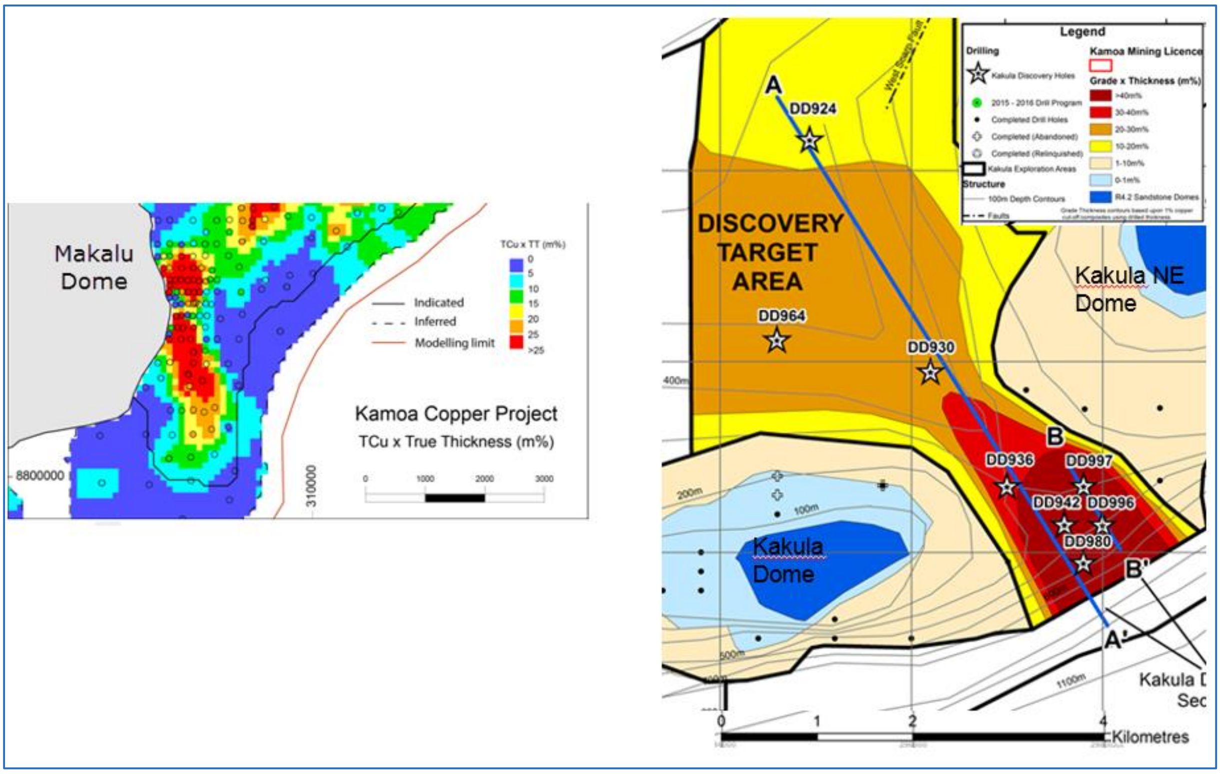


Figure provided by Ivanhoe, 2016. Scale bar on left plan map represents metres.

Additional Exploration Potential

The eastern boundary of the Mineral Resources 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 mineralization occur here, and in addition, high-grade bornite-dominant mineralization is common. Beyond these drillholes the mineralization and the deposit are untested and open to expansion, even beyond the targets for further exploration defined in Section 14.17.1 and 14.17.2.

14.18 Comments for 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 it (Datamine macros), and has validated the resource model. Amec Foster Wheeler finds the resource model to be suitable to support prefeasibility level mine planning.

The Kakula Exploration area has excellent potential for delineation of thick, continuous, high-grade mineralisation with further drilling.

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

- Commodity prices and exchange rates.
- Cut-off grades.
- Metallurgical recoveries.
  - Metallurgical testwork indicates the need for multiple grinding and flotation steps. Metallurgical variability testwork has been initiated and covered has only is in early stages and has only covered a portion of the deposit. Recent testwork has shown improved recoveries and concentrate grades compared to previous work.
- Mining plan.
  - The presence of 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. Mitigation plans for these risks could include an exploratory decline, or, potentially, a program of inclined drillholes.
  - Delineation drill programs 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.
  - Mining recovery could be lower and dilution increased where the dip locally increases on the flanks of the domes. An exploration decline is being driven which would provide an appropriate trial of the conceptual room-and-pillar mining method on the Kamoa deposit in terms of costs, dilution, and mining recovery. A decline would provide access to data and metallurgical samples at a bulk scale that cannot be collected at the scale of a drill sample. The extent of the decline and associated drifts are dependent on further studies.
- Infrastructure.
  - Exploitation will require building a greenfields project with attendant infrastructure.
- Capital and operating costs.
  - Exploitation will require building a greenfields project with attendant infrastructure.

15 MINERAL RESERVE ESTIMATES

The Kamoa 2016 PFS Mineral Reserve has been estimated by Qualified Person Bernard Peters, Technical Director – Mining, OreWin Pty Ltd. using the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves to conform tothe 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 May 2014 the Mineral Resource reported in the Kamoa 2016 PFS. The Mineral Reserve is entirely Probable Mineral Reserve that was converted from Indicated Mineral Resources.

Table 15.1 Kamoa 2016 PFS Mineral Reserve Statement

	Ore (Mt)	Cu (%)	Recovered Cu	
			(Mlb)	(kt)
Proven Mineral Reserve	–	–	–	
Probable Mineral Reserve	71.9	3.86	5,102	2,314
Mineral Reserve	71.9	3.85	5,102	2,314

1. The copper price used for calculating the financial analysis is a long term copper at $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.
2. For mine planning the copper price used to calculate block model Net Smelter Returns was $3.00/lb.
3. An elevated cut-off grade of $100.00/t NSR was used to define the stoping blocks. A marginal cut-off grade of 1% Cu was used to defined ore and waste.
4. Indicated Mineral Resources were used to report Probable Mineral Reserves.
5. The reference point for the Mineral Reserves is mill feed.
6. The Mineral Reserves reported above are not additive to the Mineral Resources.

The production plan defined for the Kamoa 2016 PFS Mineral Reserve represents the first phase of the strategy defined by the Kamoa 2013 PEA.

The mining method planned for the Kamoa Project is a combination of stepped room and pillar (SR&P) and drift and fill (D&F) and the saleable product will be copper concentrate. The processing production rate is 3 Mtpa ore. The Mineral Reserve defined in the Kamoa 2016 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 24 years. Two main areas of the ore body were targeted: the Kansoko Sud and Centrale areas.

Kamoa 2016 PFS Mineral Reserve between 170 m and 1,200 m below surface and the average dip is approximately 16 degrees. SR&P stoping has been applied to the shallower areas and D&F to the deeper areas. Although the SR&P method is simpler and has lower operating costs since no fill material is required, as the mining progresses deeper, pillar sizes increase and the lower SR&P extraction makes the cost of D&F cheaper than SR&P.

Dilution has been applied as a waste skins at the top and bottom contacts and by the use of footwall wedges below the ore body. Dilution from fill and mining losses were estimated as 5% to account for unrecovered ore.

Separate recoveries were applied to the Supergene and Hypogene metallurgical ore types. Smelter terms, constraint 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.

A break even cut-off grade of $100.00/t NSR was used to define the stoping blocks. A marginal cut-off grade of 1% Cu was used to defined ore and waste. Both the break even and marginal cut-off grade are elevated relative to the cut-off grades that can be calculated from the cost assumptions in the economic analysis of the Kamoa 2016 PFS. The process, G&A and mining costs that equate to the break even cut-off grade are approximately $60/t ore and the process and G&A costs applying to the marginal cut-off grade are approximately $20/t ore, this is equivalent to 0.8% Cu.

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 is 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 24 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 five 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.

A modification in the mining method to Controlled Convergence Room and Pillar could increase the Mineral Reserve as it may result in lower costs, through elimination of fill from some areas and an increase in extraction ratios.

16 MINING METHODS

16.1 Geotechnical

SRK Vancouver was commissioned by Ivanhoe Mines to undertake a geotechnical prefeasibility level mining geotechnical investigation and assessment and to provide guidelines for the Kamoa Copper project in the Democratic Republic of the Congo. SRK Johannesburg reviewed the work done by SRK Vancouver and provided geotechnical assistance the Kamoa 2016 PFS mining design. The primary aim of the Kamoa 2016 PFS level investigation was to increase the confidence level of the geotechnical data base as well as to undertake numerical analyses based on data from the mine site, to optimise the mine design going forward.

16.1.1 Kamoa Geotechnical Database

A substantial geotechnical data base was developed for Kamoa. The drillhole spacing over most of the initial mining area is less than 200m and in some areas it is on a 100m grid. However there is significant structural complexity in the project and further drilling will be required during the feasibility study to ensure a better understanding of this complexity. 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. During the FS an improved understanding of pillar strength is required considering that the room and pillar mining method is to be used. This will require additional testing to obtain a good understanding of the range of strength across the rock types including an understanding of the strength anisotropy. This database was used for the geotechnical design.

The assessment of the geotechnical properties assumes three primary lithological domains, namely siltstone, diamictite and sandstone (from the Roan basement rock). The ore body is located primarily within the diamictite. An additional geotechnical domain can be defined that consists of the weathered rock mass at surface. SRK Vancouver modelled the base of the weathered rock based on the weathering descriptions in the provided drill logs.

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.

These data sources give a good trend control but this leads to a low confidence on the precise location of the structures where they are not constrained by boreholes. However there is a high confidence in the structural trend. (Viljoen 2014)

16.1.2 Geotechnical Rock mass characterisation

SRK undertook a geotechnical assessment project. SRK has based all analyses on the provided drillhole database and photographs.

Primary geotechnical risks that have been focussed on during the Kamoa 2016 PFS are:

- The nature of KPS siltstone stratigraphic layer that may in parts form the hangingwall to the deposit.
- 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.

16.1.2.1 KPS Geotechnical Grade Recommended Exclusions Zones

The exclusion zones are zones of complexity (based on the drill core) where mining is assumed to be difficult (small spans and/or intensive ground support). This is based on the drillhole data and needs to be verified by the underground exposure. It is possible that mineral extraction in such zone could be potentially economical but SRK recommended for this stage of the study that mining is not considered in such areas.

In 2011, it was recommended by SRK that areas that have very poor quality rock in the back are excluded from the reserves. Also, a 20 m zone on each side of the fault should be excluded. Those “Exclusion Zones” were estimated in Table 16.1.

New information and new interpretations resulted in an overall decrease of exclusion zones in some structural domains, but a new updated structural and geotechnical study arrived with different structural and geotechnical domains, more suited to the increased data coverage. SRK presented an updated structural and geotechnical letter with numerical analysis of several variables in terms of the Kamoa room and pillar design (SRK Kamoa Design Letter, 2013).

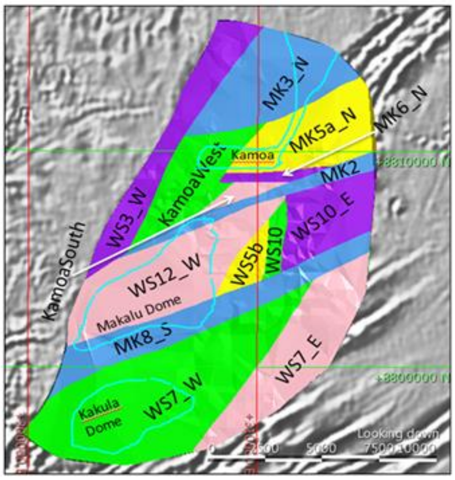
In this document different rock competency categories were updated, and there is a general decrease in the disintegrated category. A new percentage of disintegrated rocks within the new domains are illustrated in Table 16.2. The disintegrated category, together with the 20 m zone along the faults, are exclusion zones where mining might be problematic and, as suggested in 2011, that resource should be excluded from the reserves until better information is gained by underground exposure.

The other alternative is to include disintegrated rocks in the reserves and assign them a higher operating cost equivalent to the Drift & Fill (D&F) method.

Table 16.1 Recommended Exclusion Zone Percentage Ranges (April 2011)

Domain	Exclusion Zone %	Confidence Level
1	30-40	Moderate
21	30-50	Low
22	15-20	Moderate
23	20-40	Moderate
3	20-30	Low
4	20-30	Low
5	20-30	Low
6	5-20	Moderate

Table 16.2 Amount of Variable Ground Conditions in Structural Domains



	ONLY	Partly				
Domain	Fresh ONLY	Weathered	Separated	Disintegrated	Total	2010 Domains
Kamoa_S	31%	56%	31%	8%	48	Dom 22
Kamoa_W	17%	71%	53%	10%	103	Dom 1
MK2	38%	46%	23%	23%	13	Dom 23
MK3_N	11%	68%	57%	16%	44	Dom 4 + 5
MK5a_N	0%	40%	40%	40%	5	
MK6_N	9%	64%	55%	9%	11	Dom 21
MK8_S	18%	55%	45%	0%	22	
WS10	23%	38%	38%	8%	13	Dom 6
WS10_E	50%	30%	20%	10%	10	Dom 3
WS12_W	37%	52%	19%	4%	27	Dom 6
WS3_W	17%	67%	33%	17%	12	
WS5b	79%	11%	11%	0%	19	Dom 6
WS7_W	56%	24%	16%	8%	25	

16.1.2.2 Geotechnical Parameter Evaluation

SRK completed a review of all available geotechnical data within the provided drillhole database. Significant QA/QC work was completed on the data to ensure consistency and accuracy prior to rock mass characterisation.

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 (Figure 16.1). 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 RMR89 for each geotechnical domain are presented in Table 16.3.

Figure 16.1 Plan View of Three Fresh Geotechnical Domains (North, Central, South)

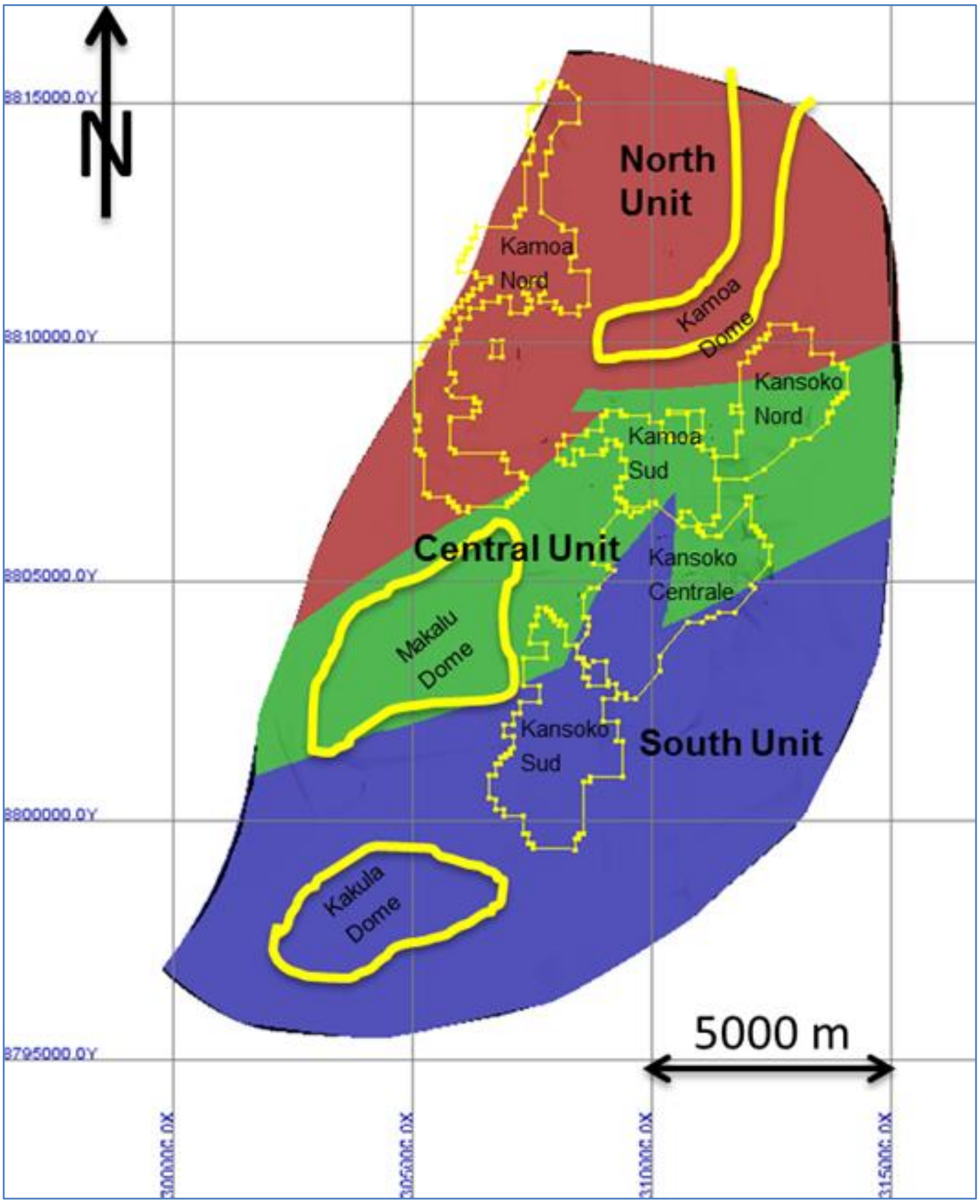


Table 16.3 Summary of Geotechnical Parameters per Geotechnical Domain

Domain	Stratigraphy	RQD (%)	FF/m	RMR_{89}	Intact Young's Modulus (GPa)	Poisson's Ratio
Moderately Weathered	KPS	40	10	43	35 Est	0.24 Est
	Diamictite	63	6	51	47	0.28
	Sandstone	55	8	48	32	0.22
Fresh, North	KPS	48	11	47	66	0.28
	Diamictite	76	4	58	67	0.27
	Sandstone	67	6	56	58	0.23
Fresh, Central	KPS	55	7	53	66	0.28
	Diamictite	73	5	60	67	0.27
	Sandstone	62	6	55	58	0.23
Fresh, South	KPS	68	8	56	66	0.28
	Diamictite	80	8	63	67	0.27
	Sandstone	71	6	59	58	0.23

The engineered intact rock strength presented in Table 16.4 considers the field estimated IRS (as logged by African Mining Consultants), field point load testing, and laboratory unconfined compressive strength testing. Table 16.4 lists the mean values, standard deviation (in brackets), and the derived engineered intact UCS of each stratigraphy within geotechnical domains. Due to a lack of data coverage across the deposit, the UCS data has been repeated in each domain for comparison to other data sources.

Table 16.4 Summary of Intact Rock Strength Estimates per Geotechnical Domain (Standard Deviation in Parenthesis)

Domain	Stratigraphy	Logged IRS (MPa)	# of UCS Tests	Intact UCS (MPa)	Point Load Test (MPa)		Engineered IRS (MPa)
					Axial	Diametral	
Weathered	KPS	45 (32)	1	123	75 (51)	61 (42)	45
	Diamictite	44 (30)	8	56 (31)	66 (54)	50 (30)	50
	Sandstone	44 (31)	4	153 (48)	88 (56)	81 (52)	75
Fresh, North	KPS	63 (47)	8	208 (36)	67 (43)	66 (44)	90
	Diamictite	72 (42)	17	98 (29)	97 (55)	71 (35)	100
	Sandstone	86 (59)	2	219 (22)	96 (55)	86 (59)	100
Fresh, Central	KPS	91 (57)	8	208 (36)	115 (54)	108 (54)	90
	Diamictite	101 (62)	17	98 (29)	80 (44)	92 (43)	100
	Sandstone	91 (63)	2	219 (22)	132 (50)	112 (60)	125
Fresh, South	KPS	116 (49)	8	208 (36)	144 (66)	140 (56)	120
	Diamictite	143 (64)	17	98 (29)	108 (48)	106 (39)	125
	Sandstone	131 (69)	2	219 (22)	121 (60)	126 (49)	125

16.1.3 Geotechnical and Structural Logging

SRK completed three site visits to the Kamoa Copper Project during 2011 for the purposes of geotechnical and structural logging QA/QC and data quality control. Ross Greenwood, Ryan Campbell, and Desiré Tshibanda completed geotechnical logging QA/QC, training, and data quality review during 22–27 June 2011. From 5–12 August 2011, Ross Greenwood and Desiré Tshibanda completed geotechnical logging QA/QC, reviewed changes implemented to logging practices, and conducted additional data quality reviews 12–17 August 2011, Wayne Barnett, SRK completed a review of structural data collection and the structural geology model, and provided input for future data collection and interpretation. The site visits and accompanying memoranda are considered to satisfy tasks within SRK's scoping level proposal issued to Ivanhoe on 29 May 2011.

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's geotechnical engineers have been recommended at all stages of data collection.
- Structural geology findings: 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.

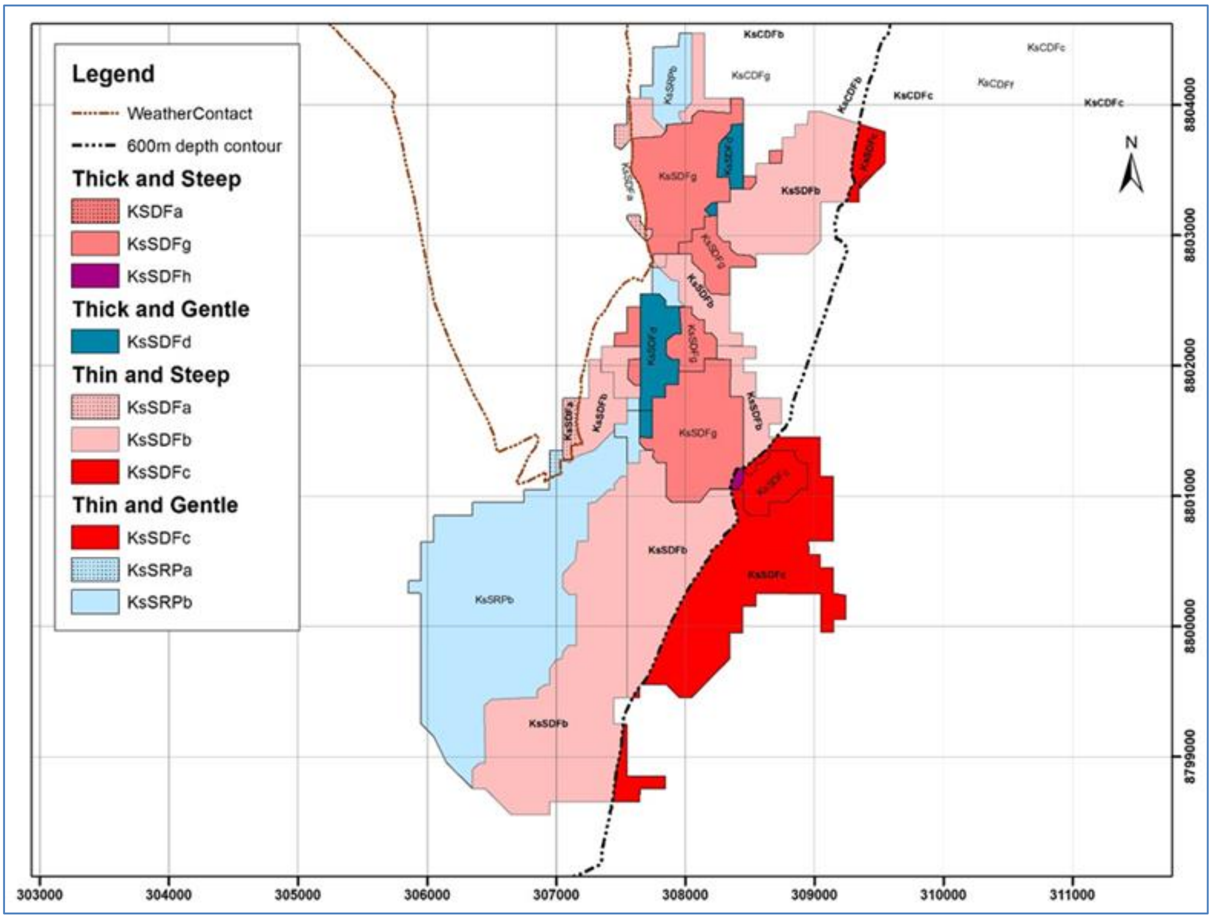
16.1.4 Geotechnical Design

Ore Body Geometry

The orebody geometry is depicted in Figure 16.2 where the different thickness and steepness is indicated. From this data it can be ascertained that the orebody dips between 0° to 34° with an average dip of 16° and the thickness between 4.6 and 5.6 m with an average thickness of 5.1 m. A number of different mining methods were assessed for their suitability in mining the Kamoa deposit. The review indicated three methods could be used:

- Normal room and pillar mining for areas with a dip of less than 12°.
- Stepped room and pillar up to a dip of 34° and a depth of less than 500 m.
- For depths in excess of 500 m the Drift and Fill mining method was chosen.

Figure 16.2 Orebody Geometry (after SRK Vancouver gentle implies shallow dipping)



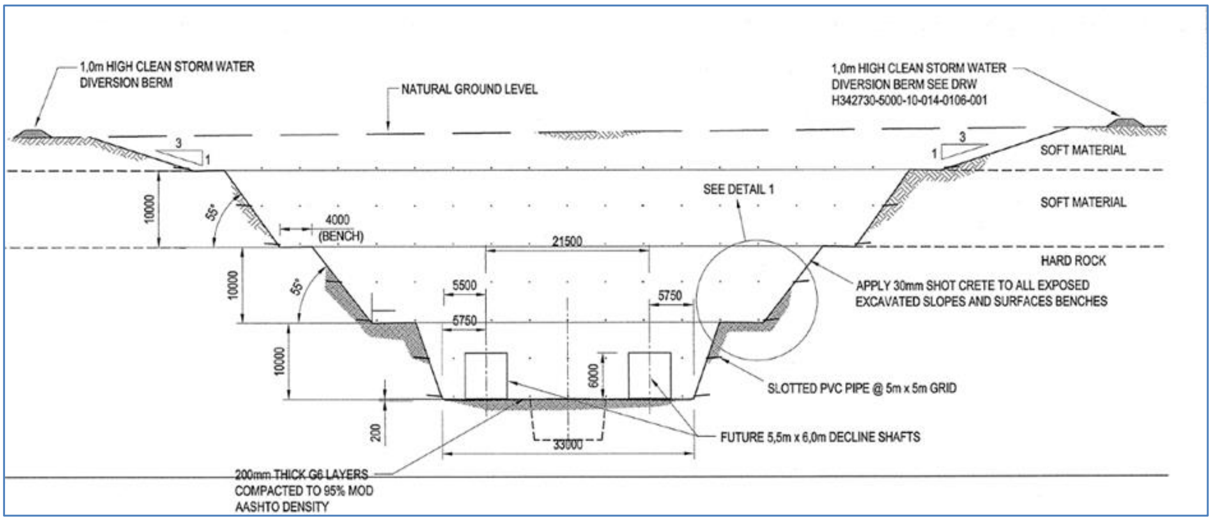
16.1.4.1 Portal Access

The access to the orebody is via a boxcut and portal that has been excavated. The boxcut is 35 m deep and has limited face heights and no slope stability risks are expected on an overall slope scale, but only on a bench scale if the recommended support is not installed. The boxcut design requirements are shown in Table 16.5 and Figure 16.3.

Table 16.5 Boxcut Design Requirements

Domain	Depth Range (m)	Bench Face Angle (°)	Max. Bench Height (m)	Berm Width (m)	Comment
Soils	0–5	30	5	4	4 m set-back at contact with Transition domain
Transition	5–15	55	10	4	BFA limited by rock mass conditions
Rock	15–35	55	10	4	BFP limited to minimise impact from J2 and J3 parallel structures

Figure 16.3 Section Through the Boxcut



Boxcut Support

The objective of the boxcut support is to accomplish the following:

- Minimise movement of material.
- Minimise exposure to rain water.
- Reduce short term deterioration of rock.
- Long term stability of the slope.

The boxcut is 35 m deep and preliminary support requirements for the portal face will be 2.4 m long 19 mm full column grouted cement grouted rebar spaced 1.0 m apart vertically and 1.0 m apart horizontally. Above and around the decline portals pre tensioned full column cement grouted 6.5 m long 25 T cable anchors are required. Osro straps will also be installed. (Greenwood R., 2014)

Portal Face Support

The portal take-off point is generally positioned in competent rock or support systems employed to provide long-term stability in weaker rock. The rock above and adjacent to the portals needs to be reinforced with 2.4 m long 19 mm diameter full column cemented rebars. Above and around the portals pre-tensioned cement grouted 6.5 m long, 25 T cable anchors. Osro straps 6.0 m long will also be used for portal support (Viljoen N 2014).

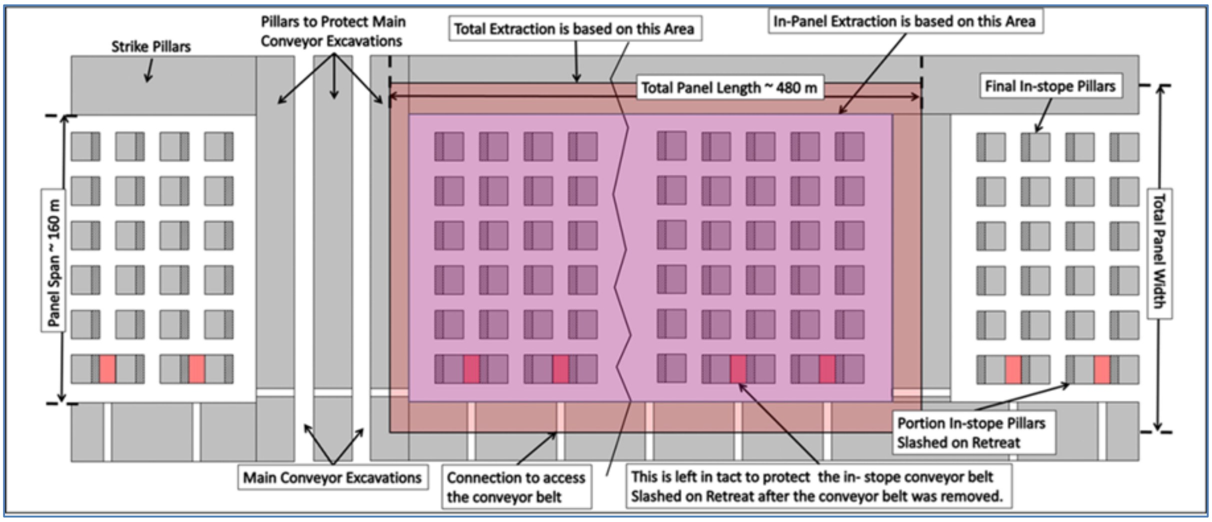
Decline Take Off Support

Excavating a large decline in unconfined weak rock in the portal will require a rigorous support system. It was proposed that the decline take off support will consist of spilling, steel sets and lagging. However the use of steel sets and lagging is not considered to be appropriate in a weak rock mass by SRK Johannesburg and an alternative support system with thick shotcrete or Armco support will be considered during a site visit to the boxcut. In addition to this a tool box of support elements including friction, rebar, cable anchors, face plates, wire mesh, straps, and shotcrete should be readily available (Viljoen N 2014).

16.1.4.2 Geotechnical Discussion of Mining Methods

Initially during the Preliminary Economic Assessment PEA it was recommended that all extraction at a depth less than 500 m and where the dip was less than 12° normal room and pillar mining would be used. The room width will not exceed 10 m. Where the dip is greater than 12° or the depth was in excess of 500 m a drift and fill method would be used. The technique is usually used for relatively flat-lying deposits. A typical room and pillar layout is shown in Figure 16.4.

Figure 16.4 Typical Room and Pillar Layout



However the majority of the ore body planned for extraction has a dip of greater than 12° the use of stepped room and pillar (SR&P) mining layout was proposed. SR&P is an adaption of room and pillar where the ore body is developed in a series of horizontal steps. Haulage ramps are designed diagonally against the dip of the ore body at an apparent dip of 8.5°.

This enables the use of mechanised equipment. Mining advances downward along the step room angle with each step having a relatively flat production floor. Furthermore the design of the roadways ensures that the pillar heights do not exceed 6 m and therefore the design criteria for the pillar will remain the same. The stability of these pillars that are not normal to the roof is not expected to be compromised however this needs to be assessed using numerical modelling during the FS.

The room width was recommended to be 10 m for the majority of the rock mass but the room width will need to be reduced to 5 m where weathering conditions (RMR>50) can be expected. The mining height for the pillar design is 6 m with 4 m considered as an alternative (width will be dependent on the grade profile and the machine height). The minimum width to height ratio for the pillars must be 1.5.

Drift and Fill

This mining method consists of access drives developed on the strike of the ore body spaced 75 m apart. Drifts with widths not exceeding 5 m are developed on an apparent dip up to 8.5° between the drives. The development of the drifts represents the 'stoping' of the operation. Two drifts are developed 5 m apart and holed into the two drives. They are then filled with paste fill and the 5 m wide drift between the 2 filled drifts is mined. This sequence is repeated as the drives advance and more drifts become available. Drift and fill mining is a flexible mining method that allows for a higher percentage recovery of the ore zones. However to reduced potentially high stress conditions regional stability will be provided by 20 m pillars placed 120 m apart on parallel to the drifts. Mining of the ore is completed with the same equipment used for mine development. During the FS further work needs to be done to assess the stability of the apexes created and the support requirements. Furthermore during the FS a bulkhead design will be required for the containment of the pastefill in the drifts.

SRK Vancouver considered that this mining method within the Kamoa context will require cemented rock fill (CRF) 4–6 MPa strength as a result of the following:

- The undertaking of expansive tabular D&F and these depths and mining heights are precedent setting.
- The paste fill option can face issues with getting it in tight to the back and the requirement for multiple bulkheads, small apex pillars and is that significantly more closure will be required before the "paste" is able to transfer the load from HW to FW.
- The CRF would be placed and then "rammer jammed" to get it tight to the back.

After discussion with Ivanhoe it was established that the aggregate that would be require is not readily available and to obtain it commercially would be prohibitively expensive. An alternative to the use of CRT involving a combination of barrier/stability pillars with paste fill to maintain stability has been proposed by SRK Johannesburg. (Murphy S., 2014)

16.1.4.3 Pillar Design

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 recommended pillar dimensions for a range of depth intervals are summarized in Table 16.6 and Table 16.7 show the stability pillar requirements. 3D numerical modelling was also used in the design process.

Table 16.6 In Panel Dimension Requirements for a Factor of Safety of 1.5

In Panel Pillars			
Mining Height (m)	**Mining Depth (m)**	**Pillar Width (m)**	**Pillar Length (m)**
6	100	9	9
6	200	10	10
6	300	14	14
6	400	17	18
6	500	21	21
6	600	24	25
4	100	6	6
4	200	8	9
4	300	11	11
4	400	14	14
4	500	16	16
4	600	19	19

Table 16.7 Stability Pillar Requirements

Stability Pillars			
Depth (m)	**Strike Pillar Width (m)**	**Dip Pillar Width (Ore Body Thickness 4 m)**	**Dip Pillar Width (Ore Body Thickness 6 m)**
<200	20	10 m	15 m
200–400	30	15 m	20 m
>400	40	20 m	30 m

The following additional pillars parameters where used for the specific situations:

- 5 m x 5 m airway slots can be cut through the dip and strike barrier pillars. The spacing between these slots should not be less than 60 m.
- Bracket pillar of 20 m either side of faults are required and will replace the stability pillars were possible.
- 20 m pillars with access slots will be left either side of the declines and main conveyor systems.

The extraction ratios obtained from the work done above are detailed in Table 16.8.

Table 16.8 Extraction Ratios

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
600	6.0	54	40	63	47
600	4.0	61	47	69	53

The extraction ratios that have been used for the Kamoa 2016 PFS estimate the total primary extraction. The slashing of in panel pillar has the potential of increasing the extraction ratio. However in this stage of the knowledge and the uncertainty about the geological and geotechnical nature of the deposit it is not recommend that this is included in any reserve estimate until a proper understanding of the ground behaviour is gained.

Design assumptions were checked using a numerical modelling form of analysis based on ground reaction curves (GRC).

Stability Pillars Combined with Backfill

A 3D numerical assessment has been carried out to assess the possibility of using paste fill and stability pillars to provide a replacement for the CRT that was recommended. The Map3D linear elastic numerical modelling package was used. The model consisted of an area of D&F mining extended from a depth of 500 m to a depth of 800 m with a length of 1,000 m. Four different models were run namely:

- CRT fill only
- Pastefill only
- Pillars only
- Pillars and pastefill

An energy release criteria was used to do a comparative assessment between these methods and it was found that for the paste fill and the pillars only scenarios the energy release rates where significantly higher than both the pillar and pastefill and the CRT fill scenarios. The results also indicate that the use of paste fill and barrier pillars results are compatible to the CRT fill results and from this it can be concluded D&F method can be implemented with strike pillars with a 20 m width every 120 m on dip.

16.1.5 Preliminary subsidence review

Background research indicated that some level of surface subsidence will occur but with stable pillars will be contained to elastic and therefore minimal. However if pillar instability does occur major subsidence could occur. Therefore additional testing will be required to better understand potential pillar behaviour. Boundary pillars and bracket pillars have also been include in the design to ensure that mitigate the risk of plug type failure, but a more detailed knowledge of the location and orientations of structures is required to understand the risk.

16.1.6 Geotechnical risks

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

- The pillar design and extraction percentages are based on the summarized data obtained from drill core only. The information is considered representative but needs to be verified through data collection from underground exposures.
- Good quality conventional blasting was assumed in SRK's analyses, 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 require a modification in the adjustment factor (80% for Poor Blasting) and the DRMS will reduce by 15%.
- Local ore body geometry is not yet adequately defined. Parts of the ore body may be complexly faulted with displacements of only a few metres. This needs to be investigated with future exposure of the rock, and may impact the nature of the chosen mining method.
- The nature and variability of the contacts of the ore body are also not clearly understood by SRK at this stage. Dilution factors used in the reserve will depend on the ability of the geologists to determine the location of ore contacts. This may also impact the nature of the chosen mining method.
- In portions of the various mining areas, the KPS siltstone 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.
- A revised stratigraphic model is required to review the amount of pyritic siltstone that could be exposed in the mining excavations. Investigations will need to be completed to characterize the rate of degradation that exposed KPS will undergo.
- The current structural model defines structures depending on the level of confidence in their existence and their expected continuity. Early stage rock exposure will help define the local continuity and geometry of the exposed fault systems, which will help characterize their impact on the rock mass conditions and groundwater permeability.
- The groundwater model for the current PFS needs to include the current defined fault network. Different groundwater transmissivities can be assumed for each type of defined structures, range from greatest transmissivities in the Domain Boundary Faults, to least transmissivities in the very low confidence fault/fracture systems.

16.1.7 Recommendations for Feasibility study

For the next stage of study the current ground conditions and structural geology assumptions must be must verified from underground exposures. SRK recommends to undertake comprehensive structural and geotechnical mapping and geotechnical instrumentation programme in the proposed.

A more focused assessment of hangingwall conditions and its implications support should be carried out.

A suitable program of stress measurements should be undertaken when underground access becomes available.

Additional laboratory rock testing should be undertaken to obtain a good understanding of the range of strength across rock types and the distribution of lower and higher strength through all the domains. Testing should also be directed at understanding strength anisotropy as well.

16.2 Underground Mining

The underground mine design, scheduling were prepared by Ivanhoe and reviewed and incorporated into the Kamoa 2016 PFS by OreWin. The Kamoa ore body geometry indicates varying ore body thicknesses and slope. The ore body dips between 0° and 34° with an average dip of 16°. The thickness varies between 4.6 m and 5.6 m with an average thickness of 5.1 m. The SR&P method was chosen for areas with dip of up to 34° and a depth of less than 500 m and at depths in excess of 500 m, the D&F mining method was chosen.

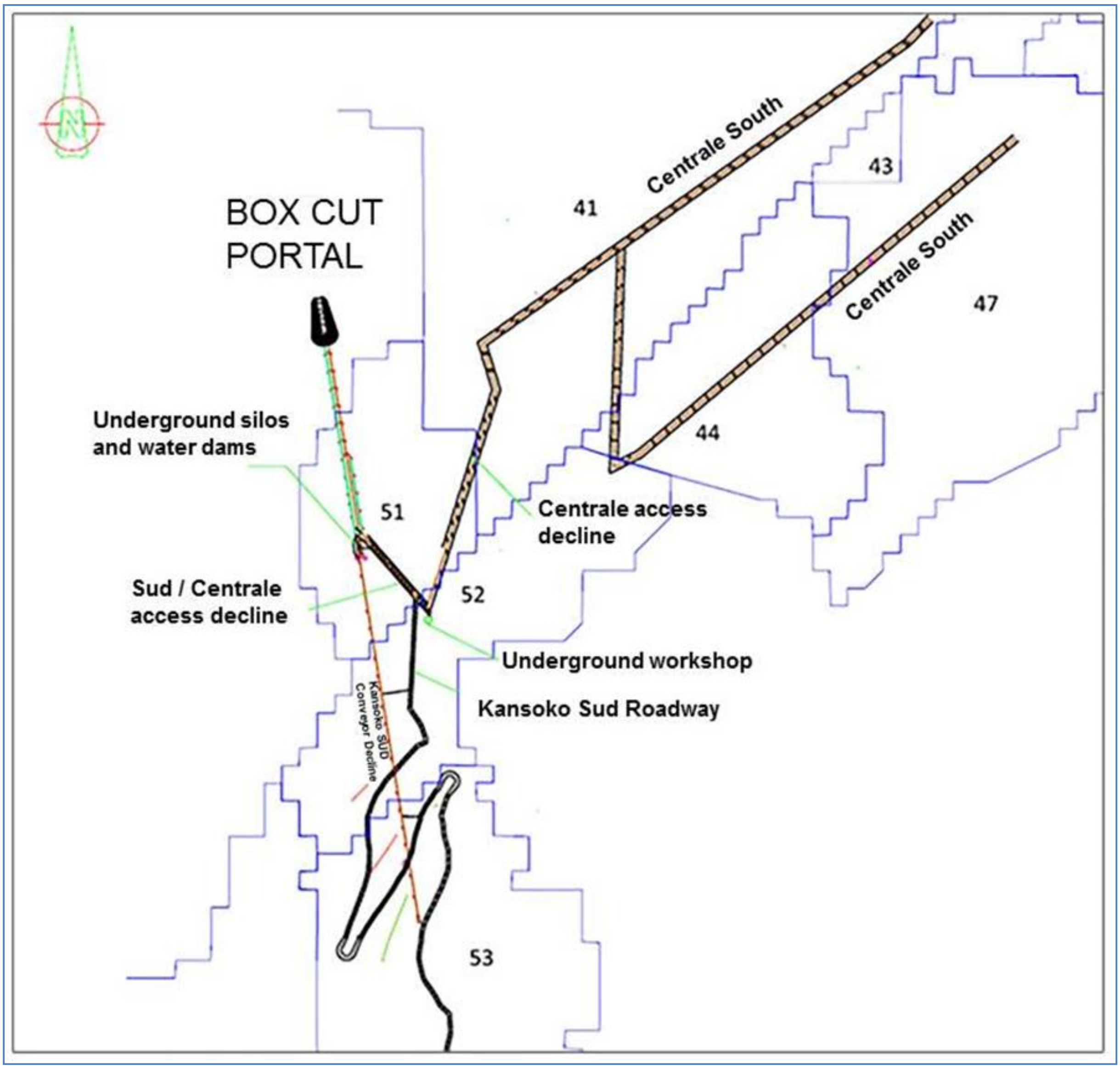
The Kamoa mine will be a mobile, trackless, mining operation. Access to the mine is planned to be via a twin decline system from Kansoko Sud portal.

The Kamoa 2016 PFS production schedule has been developed based upon a 3 Mtpa production rate and high grade scenario.

Only mining areas within a reasonable distance of the proposed access declines were included in the production schedule.

The scheduled mining areas 41, 43, 44, 47, 51, 52, and 53 are shown in Figure 16.5.

Figure 16.5 Kamoa 2016 PFS Mining Zones



16.2.1 Mining Methods

The Kamoa ore body geometry indicates a different ore body thicknesses and slope. The ore body dips between 0° and 34° with an average dip of 16°. The thickness varies between 4.6 m and 5.6 m with an average thickness of 5.1 m. The Stepped Room and Pillar method was chosen for areas with dip of up to 34° and at depths of less than 500 m and at depths in excess of 500 m, the Drift and Fill mining method was chosen.

The first ore are planned to be delivered to surface in 2016 with first stoping production planned to be achieved in 2017 and ramp-up to produce an average of 3 Mtpa. Access to the mine is planned to be via twin declines system from Kansoko Sud Portal and to support mining of Kamoa deposit total of 11,736 m of development will be undertaken.

16.2.1.1 Stepped Room and Pillar

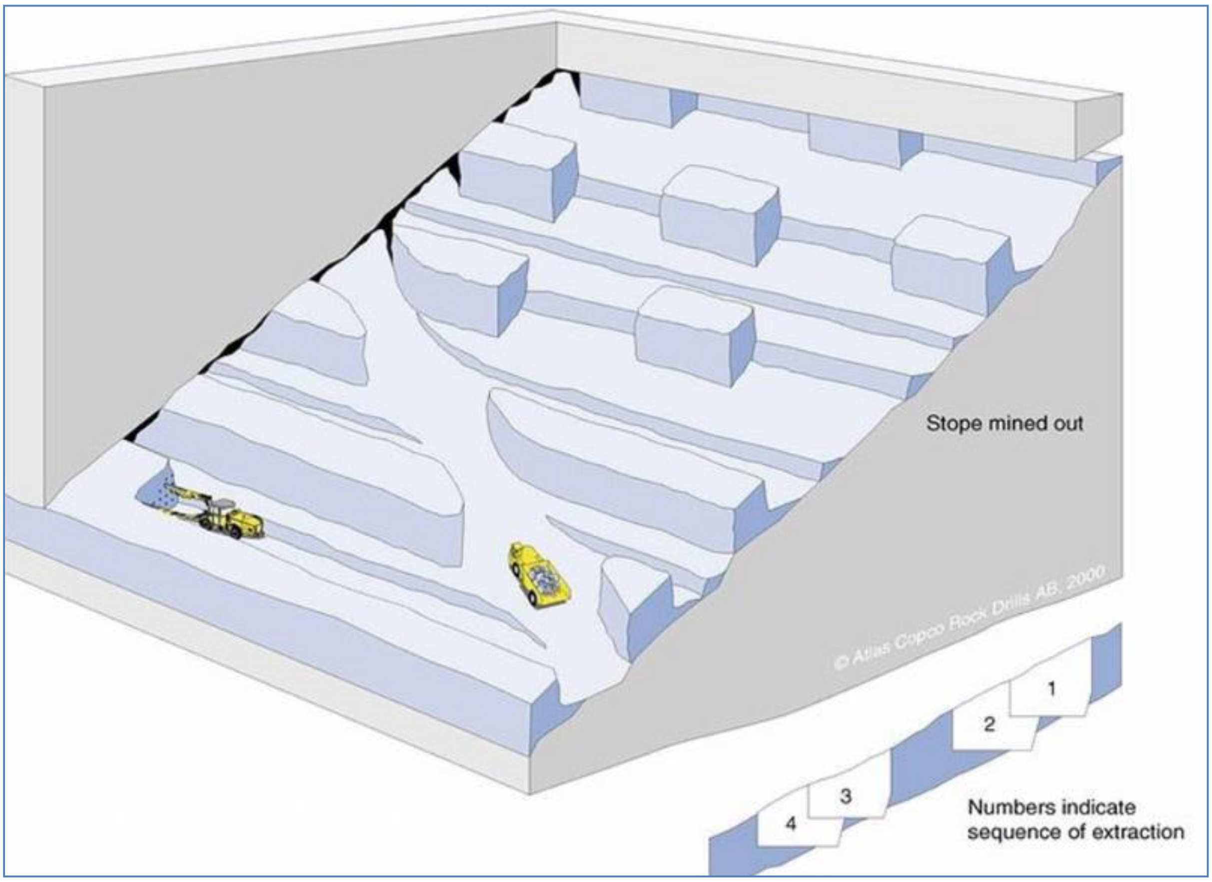
The SR&P mining method is suitable for orebodies dipping from 12° to 34° and with orebody thickness ranging between 3.5 m and 6.0 m. SR&P is an adaptation of the room and pillar mining method.

The orebody will be developed in a series of horizontal steps. Haulage ramps are mined against the dip of the orebody at an apparent dip of 8.5° which enables mechanised equipment to be used.

Mining advances downward, along the stepped room angle, with each step having a relatively flat production floor.

An example of the stepped room and pillar layout is shown in Figure 16.6.

Figure 16.6 Typical Stepped Room and Pillar Layout



The size of the remnant pillars increase with depth to accommodate the increasing stresses. Table 16.9 summarises pillar dimensions per 100 m increment in depth below surface, at a mining height of 6 m.

Table 16.9 SR&P Pillar Dimensions for a 6 m Mining Height

Max. Mining Depth (m)	Pillar Width (m)	Pillar Length (m)	Pillar Height (m)	Weight to Height Ratio	DRMS (MPa)	FOS
100	9	9	6	1.50	38	2.4
200	10	11	6	1.75	38	1.6
300	14	14	6	2.33	38	1.5
400	17	18	6	2.91	38	1.5
500	21	21	6	3.50	38	1.5
600	24	25	6	4.08	38	1.5
700	27	28	6	4.58	38	1.5
800	31	31	6	5.15	38	1.5
900	33	34	6	5.58	38	1.5
1,000	36	36	6	6.00	38	1.5

Figure 16.7 SR&P at 100 m below Surface and 6 m Mining Height

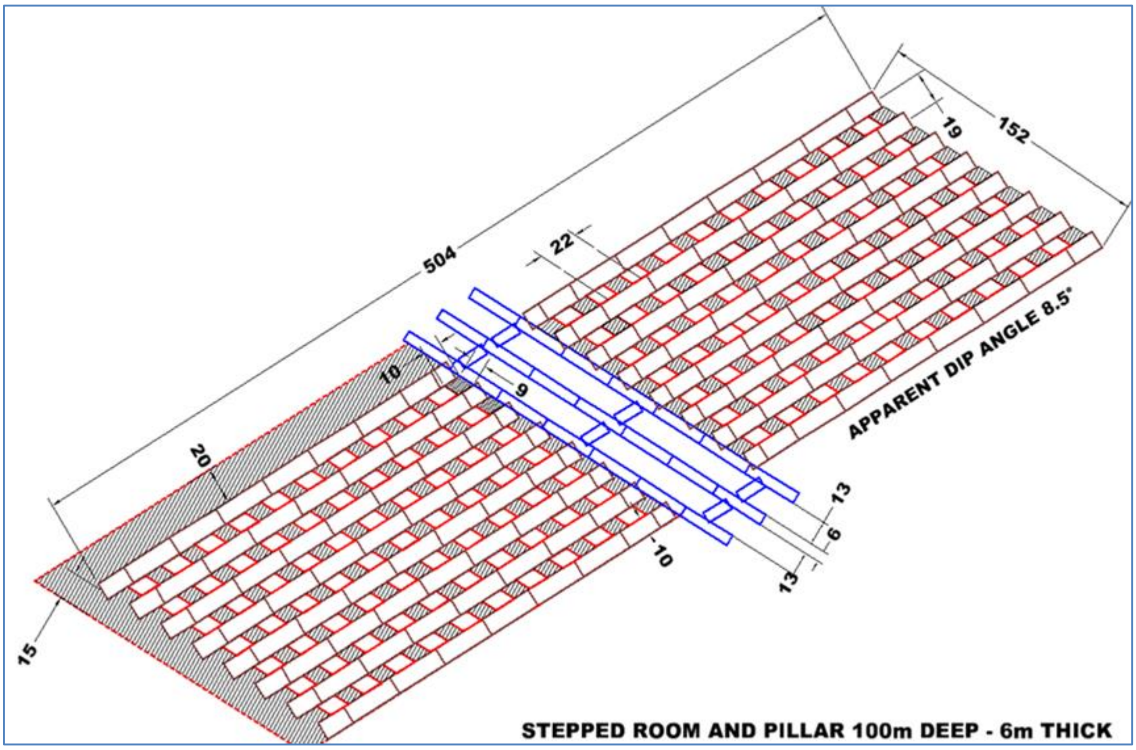


Figure 16.7 shows a proposed typical layout at a depth of 100 m and an apparent dip of 8°.

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 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 SR&P method vary with depth. These are summarized in Table 16.10 for ore body thickness of 4 m and 6 m.

Table 16.10 Calculated Extraction Ratios for SR&P 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 Drift and Fill

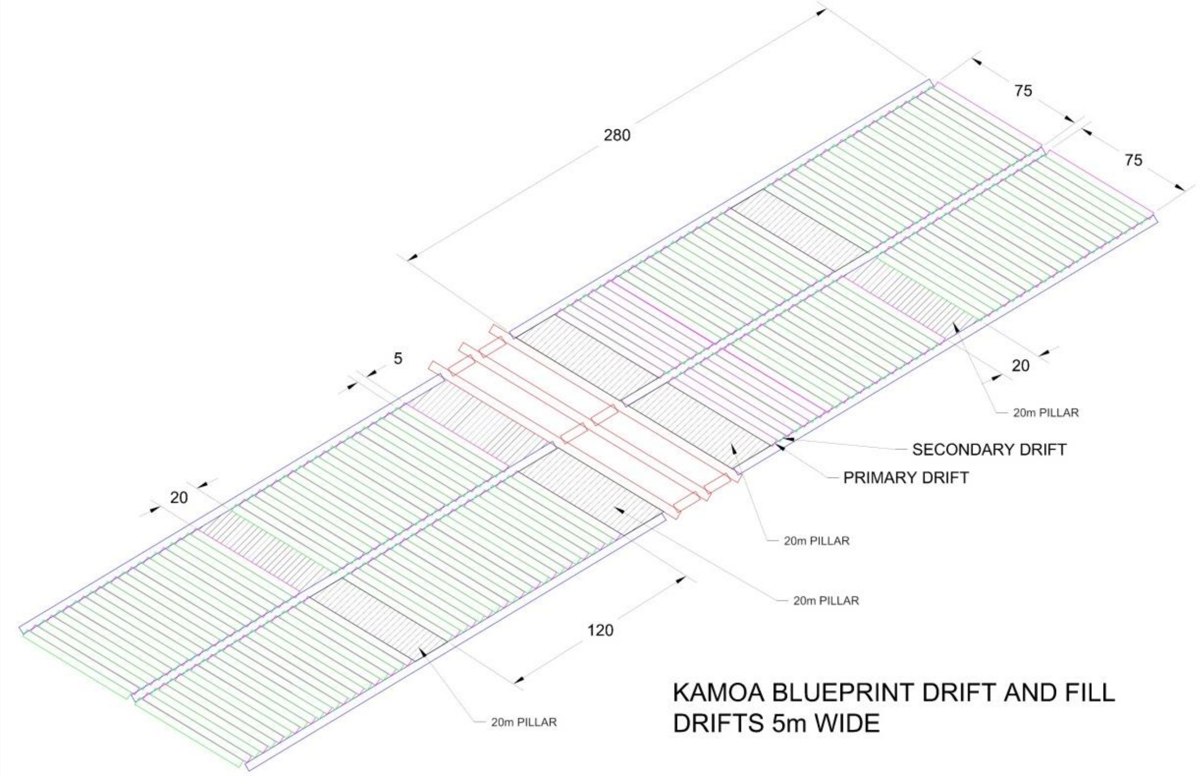
The D&F mining method involves developing access drives on the strike of the orebody and spaced 75 m apart. Parallel drifts will be developed on an apparent dip of 8.5° between the drives. The development of the drifts represents the 'stopping' operation.

Two drifts will be developed 5 m apart, these drifts are then backfilled and the 5 m wide drift between the two filled drifts is mined. This sequence will be repeated as the drives advance and more drifts become available.

The extraction ratio for areas mined using the D&F method has been calculated to be 85%.

Figure 16.8 shows the typical D&F layout.

Figure 16.8 Typical Drift & Fill Layout



16.2.2 Mining Dilution and Recovery Factors

When assessing the proposed mining methods, two sources of dilution were identified:

- A 30 cm mineralised skin in the hangingwall and 30 cm mineralised skin in the footwall.
- The wedge area that is formed as the ore body dip increases.

The 30 cm skins have been applied in the block model as independent blocks to represent over-break of either the hanging or footwall. These dilution blocks contain a representative grade and density which has been calculated based on the properties of the parent blocks.

Wedge dilution is attributed to planned, over-breaking of the footwall to ensure that the movement of mechanised equipment is at inclinations as close to horizontal as possible. The calculation of the wedge dilution is shown in Figure 16.9 and Figure 16.10.

Figure 16.9 Illustrative SR&P Planned Dilution

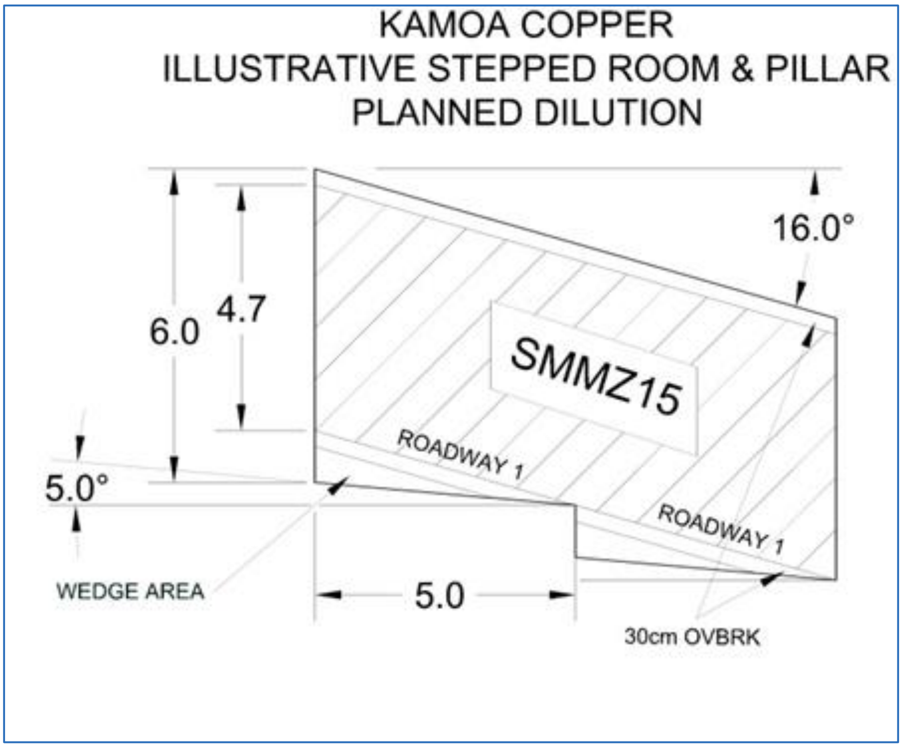
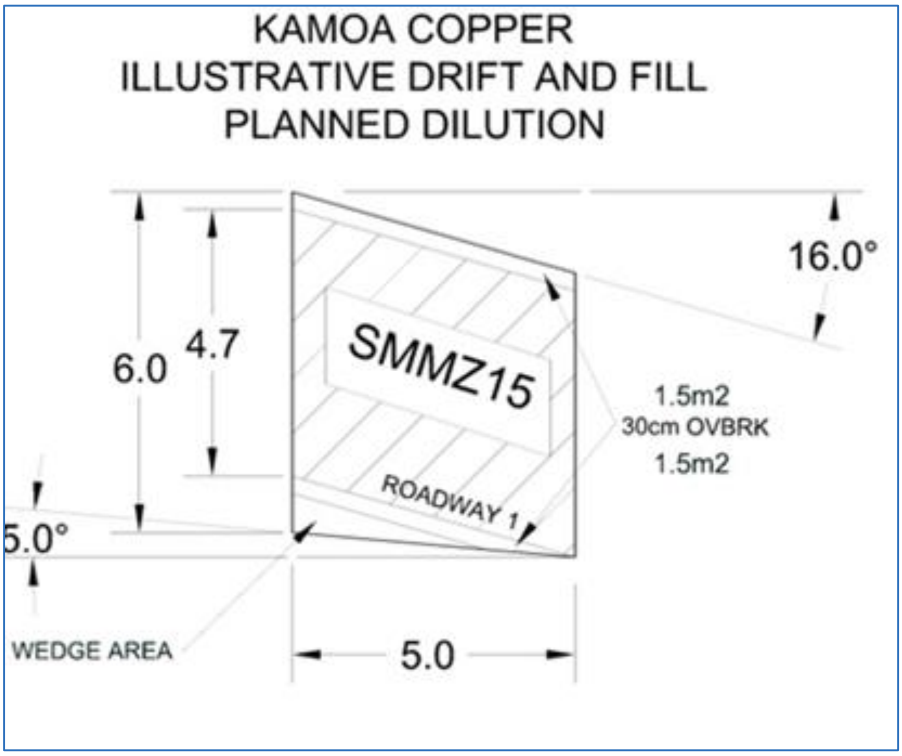


Figure 16.10 Illustrative D&F Planned Dilution



A mining loss factor of 5% has been applied to all (D&F and SR&P) mining areas. This allowance comprises:

- A mining loss of 2%. This loss is attributed to the mining method since it is expected that the mining operation cannot recover every tonne blasted.
- A loss of 3% due to geological losses.

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 which minimizes 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 developed into the Kansoko Sud mining area.

Development dimensions will be 5.5 m (W) x 6.0 m (H) based on the conveyor design, ventilation intake requirements and sizes of equipment.

Figure 16.11 shows the position of the portal in relation to underground access infrastructure.

The portal is positioned to facilitate quick access to the shallower parts of the ore body 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 16.11 Underground Access Infrastructure

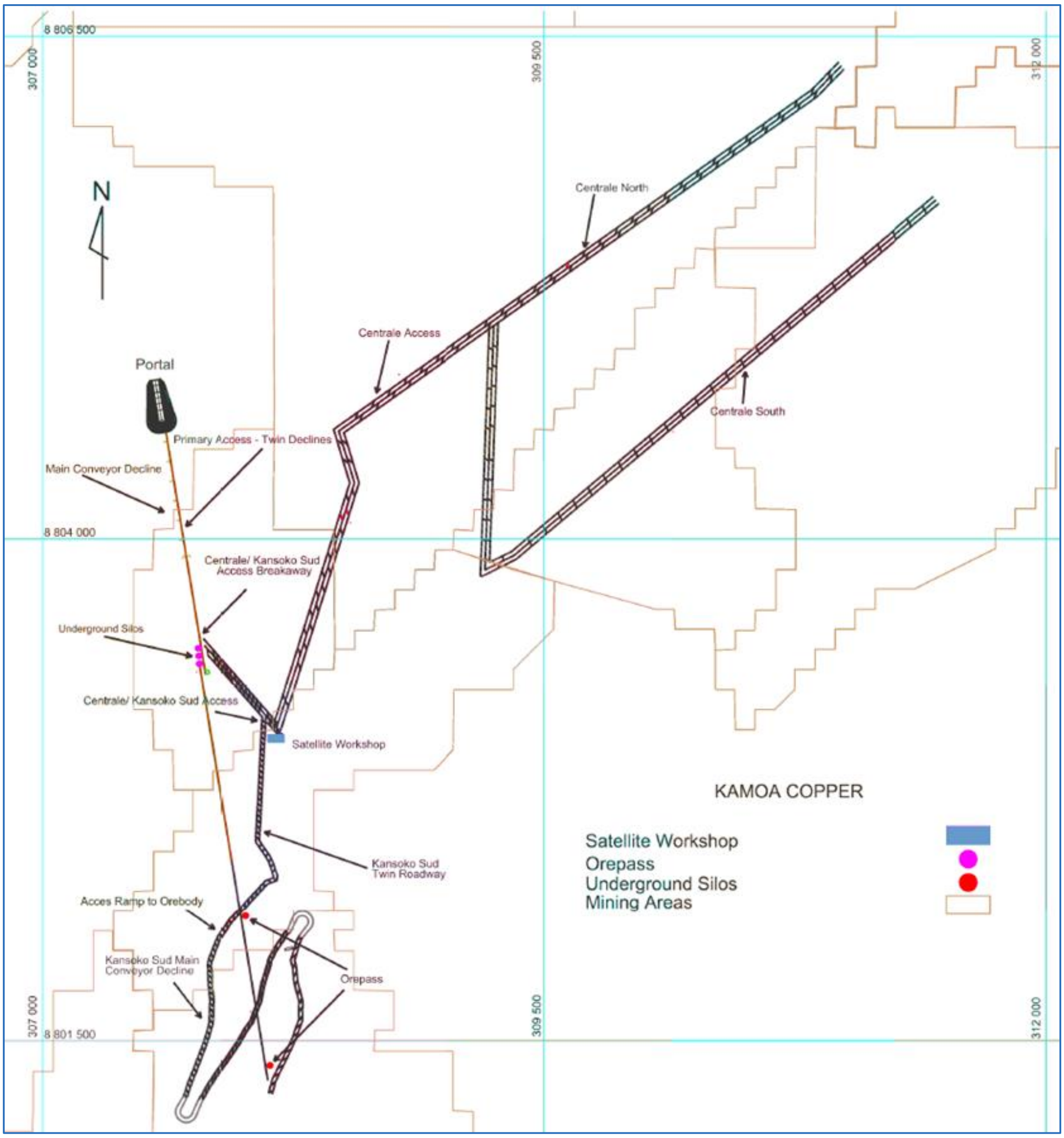
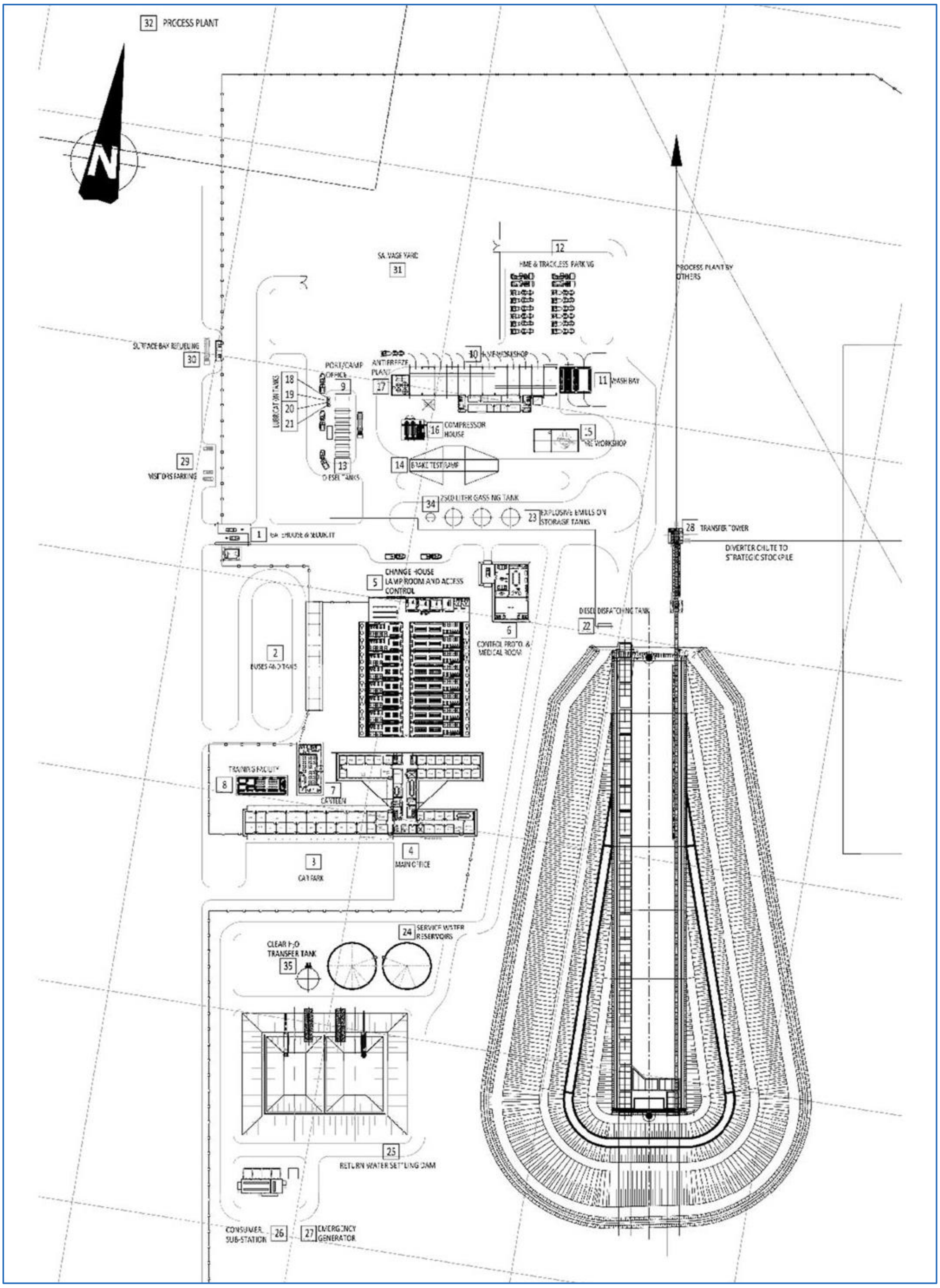


Figure 16.12 shows the position of the portal in relation to the surface infrastructure.

Figure 16.12 Portal Position in Relation to Surface Infrastructure



16.2.4 Paste Backfill

Golder Associates Ltd.'s Paste Engineering and Design Group was commissioned by Ivanhoe to prepare the engineering study for mine backfill preparation and delivery of paste backfill for the Kamoa 2016 PFS. Golder Associates were commisionsed by Ivanhoe to design the surface paste fill plant and underground distribution system. A paste fill strength of 1 MPa after 28 days was used for the D&F mining design.

Based on an evaluation of different types of backfill characteristics; i.e. strength, quantity, etc., it was concluded that Paste backfill is the viable option for the Kamoa Project. Paste backfill would serve the life-of-mine backfilling requirements and minimize the quantity of tailings reporting to the TSF. Paste backfill will only be introduced in the drift and fill mining areas, other room and pillar mining areas will not be backfilled. Various optimisations have been identified during the course of the study; the most notable is the opportunity to reduce high volumes of water being transferred to and from the backfill plant by locating the cyclones in the concentrator. As the backfill study lagged the overall study due to delays in obtaining an acceptable laboratory unconfined compressive strengths, the optimisation of the location of the cyclones could not be included, but could potentially realise a substantial saving.

16.2.4.1 Paste Backfill Testwork

Cemented paste fill testing was performed at Golder's Laboratory in Sudbury, Ontario, Canada. A number of different tailings samples were evaluated in the testing program. The final sample received, labelled as ISF4, was used in developing this prefeasibility evaluation.

The tests consisted of:

- Material characterisation;
- Settling testing;
- Filtration testing;
- Rheology; and
- Unconfined compressive strength (UCS) testing.

The testing concluded that the tailings expected to be produced by the Kamoa concentrator could be used in the formulation of paste backfill, however they had a very fine particle size distribution (PSD), and the tailings filtered poorly due to the presence of elevated concentrations of mica mineralization. The mica mineralization also required elevated addition levels of normal Portland cement to attain the target backfill UCS.

It was decided to reduce the fines content of the tailings through conditioning by hydrocyclone (cyclone) separation as well as the addition of locally available Kalahari Sand.

The underflow of the cyclone separation process provided a coarser PSD that proved to be well suited to vacuum disc filtration. It is interesting to note that the relative concentration of mica mineralization decreased slightly in the cyclone underflow compared to that of the mica in the cyclone feed.

The cyclone underflow and sand blend greatly improved the UCS results over those of the cyclone underflow alone.

16.2.4.2 Backfill Plant Design

The backfill plant design comprised of agitated storage tanks, hydrocyclones, vacuum disc filter, twin shaft high shear mixer, cement silos, thickener and various process storage vessels. The process is designed to allow the best degree of process control as is needed to maintain consistent moisture content of the backfill for the delivery of paste into the stopes.

The process is split into four main processing functions: cyclone classification and dewatering, vacuum filter dewatering, mixing of tailings and sand, and pump assisted paste distribution. In addition to these are water management, waste management, and ancillary equipment.

The paste plant process has been designed to prepare a backfill product made from cyclone mill tailings and the use of sand. The mass balance is based on the design criteria for paste having a 178 mm slump and a 7 wt% binder addition rate.

16.2.4.3 Underground Distribution System

Flow models and distribution system pressure profiles were completed for the paste distribution system required for the life-of-mine extension of the underground distribution system. Based on the results of the flow models it is estimated that a positive displacement pump will be required allowing 178 mm slump paste to be delivered to all areas underground. The flow models indicate that underground booster stations will not be required and any possible over pressurization of the pipelines can be averted by installing a surface paste delivery pipeline to supply boreholes near the north access drift, near the centroid of the ore body and the south access drift.

The positive displacement pumps deliver paste to the collars of in-service boreholes that dip at approximately 70°. In total there are 3 active boreholes planned to service the drift and fill mining area. Redundancy has been built in by providing a second standby borehole at each primary service borehole location.

The surface boreholes are fitted with a 203 mm ceramic lined casing for longevity as well as to mitigate the ingress of water into the paste which would detrimentally effect paste quality.

The main paste distribution lines underground are 203 mm X52 Schedule 80 with Victaulic couplings. Branch lines that lead from the main distribution lines into the stopes are anticipated to be a combination of 203 mm Schedule 40 and HDPE DR17. HDPE pipe is acceptable for this purpose due to the lower pressures to be encountered near the end of the distribution system, as well as for ease of pipeline installation and relocation if desired. It is important to note that this is the only acceptable use of HDPE pipe within the underground delivery system.

In the feasibility phase of the project the estimated pipeline frict ion losses should be verified through flow loop testing, and flow models adjusted accordingly. A minimum dry sample weight of 300 kg will be required to perform the flow loop testwork.

Based on the geometry and overall ore tonnage contained in each drift and fill stope a large number of stopes will be mined each year. To reduce the mining cycle time it has been assumed that prefabricated reusable barricade that can quickly be installed will be used instead of conventional shotcrete barricades which greatly increase the mining cycle time.

16.2.4.4 Further Backfill Investigation

The backfill study identified the following areas for futher investigation of the backfill design and system:

- During the laboratory assessment of the Kalahari Sand it was noted that it had a relatively low pH of 4.2 compared to the cyclone underflow at 7.4. This lower pH sand might have had a detrimental effect on the strength gain observed beyond the 14 day UCS curing values which did not increase to values normally observed to 28 days of curing. There exists the possibility that the binder, which is alkali, was acting to offset the lower pH of the sand. One way to potentially offset the lower pH of the sand would be through the addition of hydrated lime to the sand and thereby increase its pH to that of the tailings. The effects of lime addition to the sand and the resulting UCS values should be investigated in the next phase of the project.
- The current proposed design has the cyclones located at the paste plant, some 2.5 km away from the concentrator. This requires the pumping of concentrator discharge slurry at a dewatered slurry concentration (50 wt%) to the paste plant, the delivery of large quantities of dilution water from the concentrator, in order to provide a suitable slurry to the cyclones and achieve optimum separation efficiency (15 wt%) and the pumping of large volumes of dewatered cyclone overflow to the concentrator.
- The pumping required in the proposed design will increase both the capital and operating costs of the paste preparation system as a result of the requirement for:
  - A thickener to dewater the cyclone overflow
  - Additional slurry and fresh water pumps;
  - Large agitated tanks;

- Large pipelines; and
- Higher energy consumption.

- The possibility exists to prepare the classified cyclone underflow at the concentrator and pump it to the paste plant when required thereby reducing overall costs. This approach merits further study in the next phase of the project.
- Reassessing the location of the paste plant in order to optimize UDS system pressures and paste retention time, following completion of the flow loop tests.
- The binder consumption for structural fill is on the upper end of the scale for paste backfill based on the target UCS of 1 MPa. Based on Golder's experience, a sizable reduction in binder requirement may be realized through either the addition of a larger proportion of sand or the addition of crushed rock into the paste matrix. The net benefits of this could also be determined if a new sample is to be tested.
- The pipeline friction losses applied in the flow model evaluation have been estimated based on the results of the laboratory testing program on the Kamoa tailings and Golder's database of tailings with similar properties. It is recommended that flow loop testing be conducted in the next phase of the Kamoa project, when additional tailings samples are available, in order to better quantify the anticipated pipeline friction losses and provide a better understanding of the underground distribution system design characteristics.

16.2.5 Underground Infrastructure

Services

Each of the declines will initially be developed using temporary services which will include a raw water column (150 mm diameter), a pump column (200 mm diameter), a 550 VAC power supply and a forced ventilation duct (1,100 m diameter).

As the decline progresses, an equipping crew will replace the temporary services with permanent equipment.

Stockpiles

Stockpiles will be excavated into the sidewall of the decline every 100 m. The stockpiles have multiple uses. Their primary purpose is to allow the efficient turnaround of the decline face in the mining cycle, allowing the removal and stockpiling of broken rock from the decline face. They will also be used when loading trucks and later as dams, substations, refuge stations and pump stations. Stockpiles will be no less than 15 m long.

Access Ramps

Access ramps have been designed to link the Kansoko Sud twin roadway system to the Kansoko Sud conveyor decline. The purpose of these ramps is to allow the access of personnel and material from the Kansoko Sud conveyor decline onto the ore horizon. In addition they act as intakes or return air-ways.

Ventilation Raises

Upcast and downcast vertical ventilation raises have been planned for the ventilation system.

16.2.6 Ore and Waste Handling Systems

Bulk transport of the ore and waste will be by means of a network of decline conveyor belt systems.

In the Centrale mining section, section conveyors will be used to transport ore from the mining face onto decline conveyors.

In the Kansoko Sud mining section, dump trucks will transport ore from the mining face into ore passes which feed onto the decline conveyors.

Both mining sections decline conveyor will convey ore into the underground storage silos for conveying to surface and onto the feed conveyor to the process plant.

Decline and Conveying Facilities

A trade-off study comparing trucking and conveying options confirmed that conveyors would be less expensive than trucking for a large, long duration mining operation such as the Kamoa project.

Trucks will be used during the development of the conveyor and service declines until the main decline conveyor and silos have been commissioned.

All decline conveyors have been designed to convey the total mine ore production of 3 Mtpa. The conveyor system design capacity was then increased by 15% to allow for waste rock handling and a further 10% to allow for production “catch up” and/or future increases in production.

As a result, all conveyors will have an operating capacity of 876 t/h and a belt speed not exceeding 2.5 m/s. In order to reduce the start-up torque, the conveyors will be started Direct on Line (DOL) with a torque limiting soft start coupling.

One of the main design considerations for the conveyor system was standardization to reduce the spares holding requirements.

The Kansoko Sud conveyor system network will consist of four decline conveyor systems.

Main Decline Conveyor No. 1

Main decline conveyor No. 1 will transport the total mine production of 3 Mtpa from the four underground silos to surface. The conveyor head pulley will discharge into a splitter discharge chute for the transfer of material onto the process plant feed conveyor or onto a ground stockpile for loading and transporting to a strategic ore stockpile.

The main decline conveyor No. 1 will be 1,100 m long, 1,050 mm wide and fitted with a class 2500 steel cord belt.

The conveyor belt drive will consist of three 250 kW units, two primaries and one secondary, mounted in a drive arrangement on the footwall, behind the head pulley and splitter discharge chute.

A horizontal, gravity-type take-up will be installed, mounted directly behind the drive arrangement.

Kansoko Sud Decline Conveyors No. 1, 2, and 3

The geometry of the Kansoko Sud ore body, with varying dips in two planes, does not allow the use of section conveyors.

In the Kansoko Sud mining section, LHDs will load ore from the mining face on to dump trucks. The ore is then trucked to ore passes which feed onto the Kansoko Sud decline conveyor system.

The dump truck tip arrangement at the top of the ore passes will include a static grizzly and a hydraulic breaker to break all oversize material.

The Kansoko Sud decline conveyors No. 1, 2, and 3, operating in series, will transfer material from the ore passes into the four ore storage silos.

The conveyors will be constructed on a modular basis for ease of installation and will be 1,050 mm wide and fitted with class 2000 fabric belt. For standardization purposes the conveyors will be of similar length.

The conveyors will be driven by two 250 kW drive units, one primary and one secondary, mounted in a drive arrangement directly behind the head pulley.

Horizontal, electric, winch take-ups will be used which will be mounted directly behind the drive arrangement. The take-up winches will be started and stopped locally and a load cell will be used to set the correct belt operating tension.

Kansoko Sud/Centrale Decline Access Conveyor No. 1

This conveyor will transfer ore from the Centrale mining area into the four ore silos. The conveyor will have the same design criteria as the Kansoko Sud decline conveyors No. 1, 2, and 3.

Centrale Conveying Systems

The Centrale conveying system consists of ten decline conveyors operating in series transferring ore from the Centrale South and North mining areas onto the Kansoko Sud/Centrale decline access conveyor No. 1.

The conveyors will have the same design criteria as the Kansoko Sud decline conveyors No. 1, 2, and 3 in order to standardize on conveyor drive units and pulleys. Where practical, the conveyors will be of similar length.

Section Conveyors

The geometry of the Centrale ore body is more uniform allowing the use of section conveyors. In the Centrale mining section, LHDs will load ore from the face onto the section conveyors which then transport ore onto the Centrale conveying system.

The section conveyor tip arrangement will consist of a skid mounted structure supporting chute work, a static grizzly, a vibrating feeder and a hydraulic rock breaker. The LHD will tip into the chute work surrounding the static grizzly which will prevent oversized material from passing onto the vibrating feeder. The grizzly will have 250 mm square apertures. A vibrating feeder below the grizzly will control the flow rate of material and discharges directly onto the section conveyors. The hydraulic rock breaker arrangement mounted to the structure will break oversize material caught on the grizzly.

As mining advances, the entire tip structure will be dragged by a LHD to a new position, closer to the mining face, and the section conveyor will be extended to suit.

The section conveyors will be constructed on a modular basis for ease of installation, extension and relocation as the mining faces advance. Access drives up to 1,000 m in length will require section conveyors designed for extending up to 1,000 m.

The section conveyors will be 1,050 mm wide, class 800 fabric belts. The section conveyors will each be installed with a 132 kW drive unit mounted on the head pulley.

The section conveyors will be started DOL with a torque-limiting soft start coupling to reduce the start-up torque. Horizontal electric winch take-ups will be used and will be mounted behind the drive arrangement. The take-up winches will be started and stopped locally and load cells will be used to set the correct belt operating tension.

Underground Storage Silos

Production from the Kansoko Sud and Centrale mining areas will be fed into four vertical ore storage silos positioned at the bottom of the main decline conveyor No. 1.

Ore conveyed from the Kansoko Sud and Centrale mining areas will feed onto a tripper conveyor installed on top of the silos.

A dump truck tip arrangement will be available, complete with grizzly and rock breaker, at the top of one of the silos. This will allow the transport of rock from the development of the Kansoko Sud decline and ore from initial SR&P mining to be trucked to the silos.

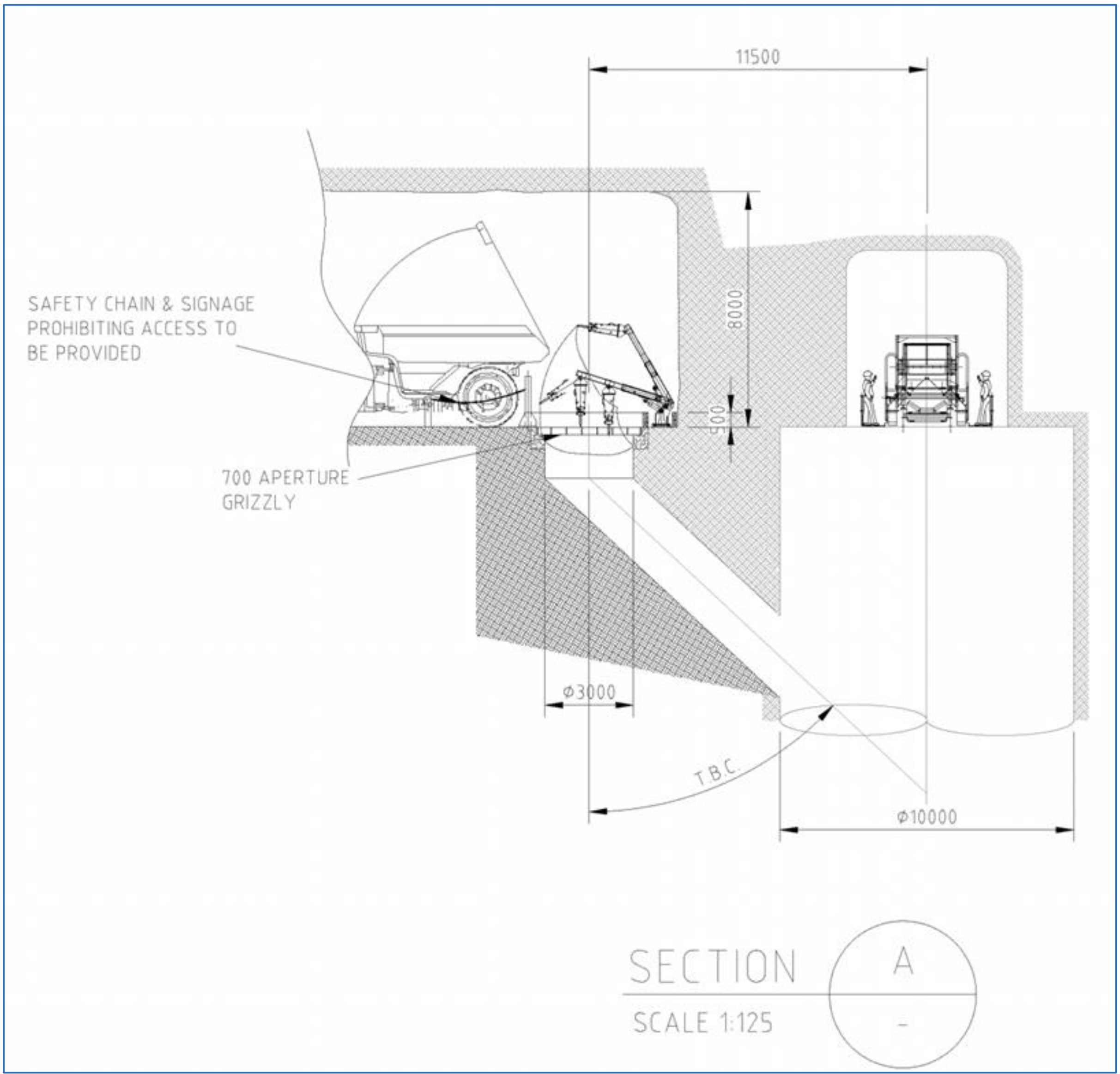
A typical dump truck silo tip arrangement is shown in Figure 16.13.

The silos will control and regulate the feed onto the main decline conveyor No. 1 and will also provide surge capacity if the main decline conveyor No.1 is inoperable due to maintenance or failure of the conveyor belt.

The storage capacity of the silos has been designed to store one day's production of 8,500 t.

The silos will be vertical with typical dimensions of 10.0 m diameter x 25 m (H).

Figure 16.13 Typical Dump Truck Silo Tip Arrangement



Ore and Waste Passes

Ore passes will be used to transfer ore from development and stoping in the Kansoko Sud mining area onto the Kansoko Sud decline conveyors No. 1, 2, and 3.

The ore passes will be located strategically within the Kansoko Sud mining area with typical dimensions of 2.1 m Ø x 30.0 m (H).

The truck tip arrangement on top of the ore pass will include a static grizzly which will prevent oversized material from entering the ore pass. The grizzly will have 250 mm square apertures. A hydraulic rock breaker will be used to break any oversize material caught on the grizzly.

The arrangement at the bottom of each ore pass will consist of outlet chute and radial doors to retain and direct material out of the ore pass. The outlet chute will feed material onto a vibrating feeder which will control the flow rate of material and discharge directly onto the Kansoko Sud decline conveyors.

Crushing Facilities

There will be no underground rock crushing facilities. A trade-off study compared underground crushers with hydraulic rock breakers found that hydraulic rock breakers were preferable, due to lower capital and operating costs and the ease of relocation in line with the advance of the mining face.

16.2.7 Mine Services and Support Infrastructure Design

The main ventilation infrastructure will include main surface fans and the refrigeration and cooling system. The main ventilation infrastructure will include the fan station at the top of ventilation raise 1S. The fan installation will be a trifurcated unit, initially only one fan will operate and the other two drifts will be blanked off.

There is allowance in the ventilation design for secondary ventilation equipment including fans, duct, stoppings and auxiliary equipment.

16.2.7.1 Mine Ventilation and Cooling Design

Ventilation Model

The quantity of ventilation air required is driven by the need to dilute diesel exhaust emissions. Using the factor of 0.06 $m^3/s/kW$ total rated power at the point of use, and based on the size and deployment of the diesel vehicles, the total required ventilation flow rate is 950 m^3/s.

Depending on the geothermal properties of the rock mass, the heat flowing into the mine from rock, combined with other heat sources and the heat from operating diesel equipment must be cooled by the ventilation air used to dilute exhaust emissions. If the generally ambient temperature of the ventilation air alone cannot adequately cool the mine additional mechanical cooling is required.

Modelling of ventilation and heat flow (using VUMA3D) confirmed that at depth in the Centrale South section, additional cooling will be required to ensure that workplace wet bulb temperatures do not exceed the design limit of 29°C.

Intake Ventilation

The intake ventilation system combines the main service and conveyor declines with two strategic fresh air raises. The raises will be 5.0 m diameter and will supply the bulk of the ventilation to on-reef airways which in turn distribute air to active areas in the Centrale and Kansoko Sud mining areas.

The first raise (2S) will be located between the Kansoko Sud and the Centrale mining areas. The second raise (5C) will be located in the Centrale South mining area.

Return Ventilation

Exhaust air will return, through airways established in worked out areas, back to up-cast ventilation raises located at the top of each mining area. Six return ventilation raises will be required, four at 3.5 m diameter and two at 5.0 m diameter.

In addition, a 2.0 m diameter ventilation raise has been planned to independently return air from the underground satellite workshop.

The up-cast ventilation raises will be equipped with surface fans.

In the SR&P sections, return airways to the up-cast ventilation raises will typically consist of two mined-out roads. In the D&F sections, strategic drifts will be kept open (not backfilled) to ensure that a route is established between the in-take system and return system. In this case, pillars will be left to create the airway and the pillars can be extracted once the ventilation district is no longer in use.

Air Cooling and Refrigeration

Ventilation modelling indicates that cooling will be required from Year 7 onwards. For practical reasons, the refrigeration and cooling plant will be located on surface.

The refrigeration and cooling plant will comprise a 7.2 MW bulk air cooler (BAC), a 7.0 MW refrigeration machine, a plant building, an 8.8 MW cooling tower, water circulating pumps and electrical and control systems.

The BAC will cool 300 m^3/s of intake air that travels down the 5.0 m diameter ventilation raise 5C feeding the Centrale South mining area.

Absorbed electrical power for ventilation and cooling will increase steadily from about 1.3 MW in Year 2 (ventilation fans) to approximately 7.9 MW after Year 7 (refrigeration plant) and at steady state production. The power requirements at steady state demand will be:

- Main surface fans: 4.0 MW
- Refrigeration plant: 1.9 MW
- Secondary ventilation: 2.0 MW

16.2.7.2 Permanent Infrastructure Design

Surface Heavy Mobile Equipment Workshop

The surface HME workshop will consist of three bays with a raised service ramp, six service bays at ground level, two "clean" bays dedicated for repair and maintenance of hydraulic components and a 20 t and 10 t overhead travelling cranes.

Mobile production support vehicles that travel to surface on a regular basis will be serviced and maintained in a surface HME workshop.

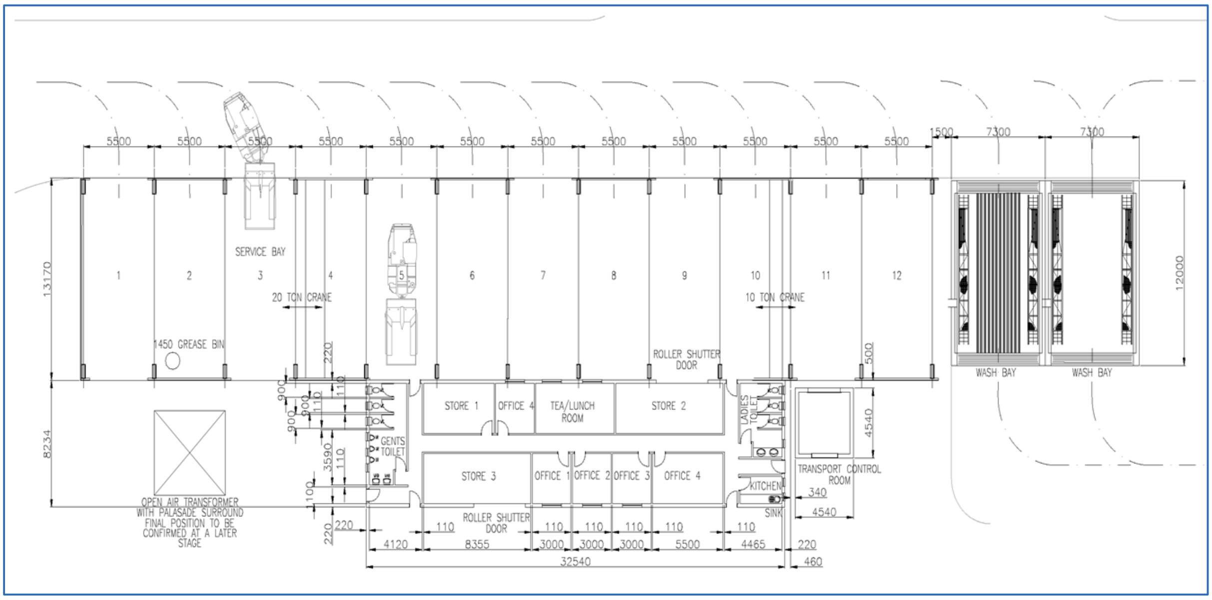
Major services required by the underground production fleet including component replacements (engines/transmissions/centre portions) will be carried out in the HME workshop.

A separate washing facility adjacent to the HME workshop will be utilised for washing light and heavy vehicles. The washing facility will be a high pressure water system complete with a dedicated silt and oil trap. Contaminated water will be pumped to the pollution control dam.

The HME workshop will include two 14 m² offices, one 26 m² store, toilets and an oil separator.

Figure 16.14 shows the plan of the surface HME workshop.

Figure 16.14 Surface HME Workshop Layout



Surface Tyre Workshop

Wheel assemblies, rims and tyres will be replaced and serviced at the surface tyre workshop. A mobile tyre handler will be used to remove and refit wheel assemblies to all mobile equipment. Wheel assemblies, tyres and rims will be stored at the tyre workshop. Wheel assemblies complete with tyres will be transported underground to the underground satellite workshop. Tyres will not be removed and fitted to wheel rims underground.

Underground Satellite Workshop

Daily and weekly servicing of the underground production fleet will be carried out in the underground satellite workshop.

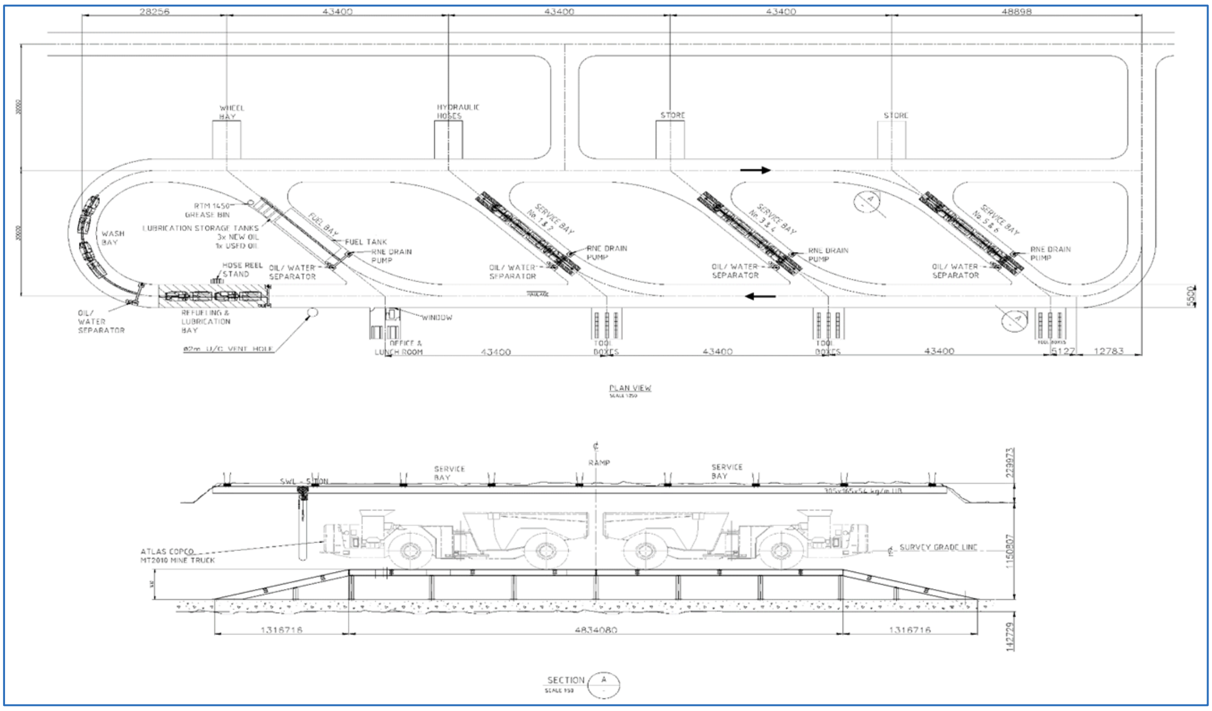
The workshop will be developed outside the mining footprint, centrally located to service production fleets for both the Kansoko Sud and Centrale mining areas.

The workshop will consist of six service bays. Each service bay will be equipped with elevated ramps with an inspection walkway in the centre. The bay will include a 10 t crawl beam fixed to the hangingwall to allow the removal and refitting of vehicle components.

The workshop will be laid out and equipped with office and lunch room, stores for equipment spares, wheel storage bay, mobile wheel handler, refuelling and lubrication bay, compressor and air receiver, two equipment wash bays, silt traps and oil separators and automatic fire water sprinkler system.

Figure 16.15 shows the plan of the underground satellite workshop.

Figure 16.15 Underground Satellite Workshop



Explosives

Explosive supplier AEL Mining Services currently manufactures bulk emulsion explosives at a plant 80 km from Kamoa. The explosives are distributed from AEL Mining Services' Kolwezi distribution hub situated 40 km from the mine. The DRC government requires all road deliveries to be escorted by the army.

At steady state production the mine will consume 10 t of emulsion per day. Allowing for a 10 day cover, a 100 t storage facility will be required on the surface, to store emulsion explosive deliveries.

AEL Mining Services will deliver explosives ancillaries by road from their magazines situated at Likasi (240 km from Kamoa). The explosives ancillaries will be transported underground to an explosives magazine, by an explosives utility vehicle.

The underground magazine will be constructed to comply with the DRC Explosives Act 26 of 1956. It will be situated, in a position central to the Kansoko Sud and Centrale mining areas.

During the initial stages of the mine's development and until the underground explosives magazine has been constructed, a temporary portable containerised explosives magazine will be commissioned on surface.

Mine Administration Office Areas

The mine administration offices, change house, canteen and training facility will be located at the portal but outside of the mining area security fence. These buildings will be pre-fabricated and designed to accommodate both the mining and process plant personnel as summarised in Table 16.11.

Table 16.11 Offices Located at the Portal

Office Building	Capacity	Facilities
Mine Administration Office	61 Employees	Open plan area with 51 offices, 3 meeting rooms, toilets, kitchenette
Canteen	150 Employees at one sitting	–
Training Facility	100 Employee	A single lecture room
Change House	To accommodate 1000 Employees	Laundry, Cap lamp and Self-contained rescue pack room
Medical Facility		A separate consulting area and a stabilisation ward.

Concrete and Shotcrete Facility and Distribution

Concrete and shotcrete material will be delivered to the mine pre-bagged and stored suitably on surface for transport underground when required.

There will be no shotcrete or concrete batching plant at the mine.

Refuge Stations

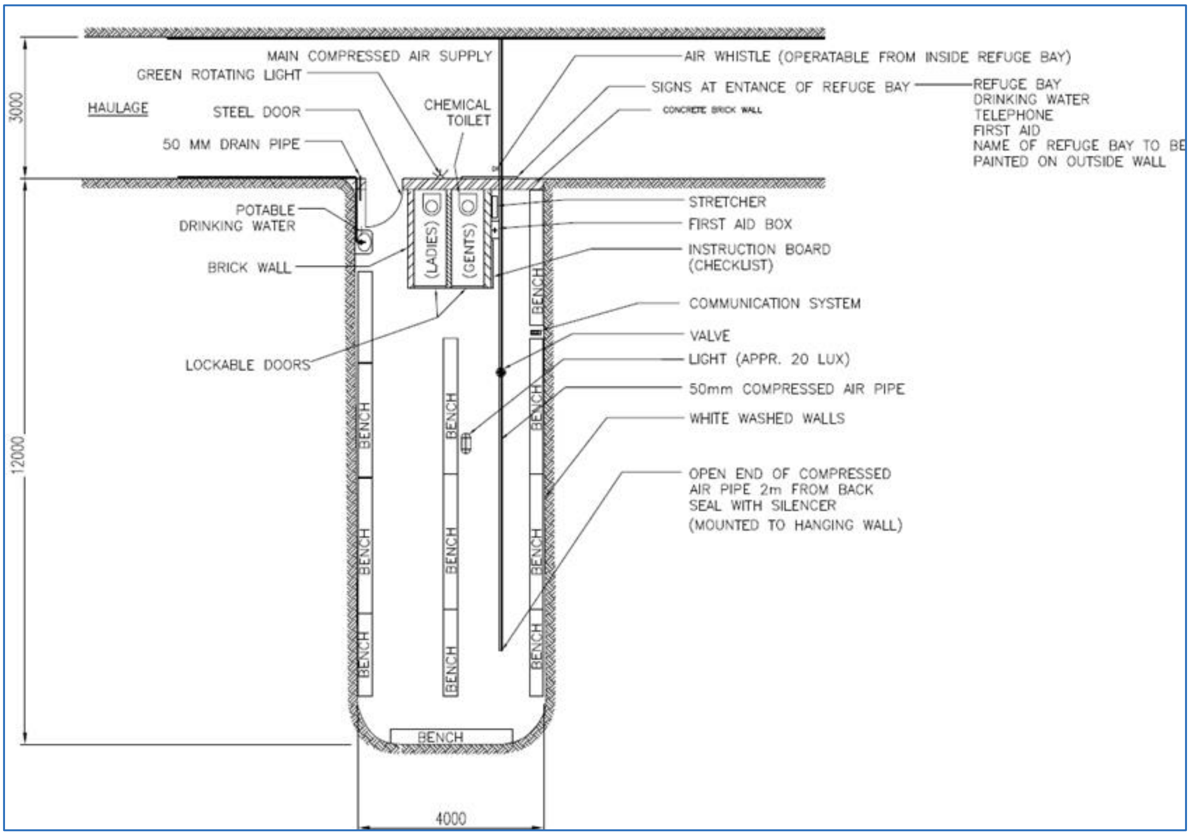
Refuge stations are required to be within easy reach of underground personnel and within the limits of protection afforded by self-contained self-rescuers.

Permanent refuge stations will be placed not more than 500 m apart with the last refuge station within 500 m of the last working place. The main decline system will be equipped with permanent refuge stations ventilated by compressed air from the compressed air station on surface.

The refuge stations will be sized based on the industry norm of 0.75 m² per person. The stations will be equipped with chemical toilets, a supply of potable water, a first aid kit including water, a communication system, and adequate bench seating.

A typical permanent refuge chamber, shown in Figure 16.16, will accommodate approximately 60 people.

Figure 16.16 Typical Permanent Refuge Station



Self-contained mobile refuge stations capable of accommodating 16 people for 36 hours will be used in the mining areas.

When the mine reaches steady state there will be 21 permanent and four mobile self-contained refuge stations.

Underground Toilets

Portable chemical toilets will be used underground and will be strategically placed throughout the mine. When full, the chemical toilet pails will be transported to surface and the contents disposed of at the mine's sewage works.

Compressed Air System

Air compressor station located on surface will provide compressed air to ventilate permanent decline refuge stations and to inflate vehicle tyres and operate pneumatic tools in the surface HME and tyre workshops.

The compressed air supply line to the permanent refuge stations will be installed in the service decline.

The compressor station will be connected to the mine's emergency power supply ensuring that compressed air is supplied to the refuge bays during power outages.

Fuel and Lubricant Distribution

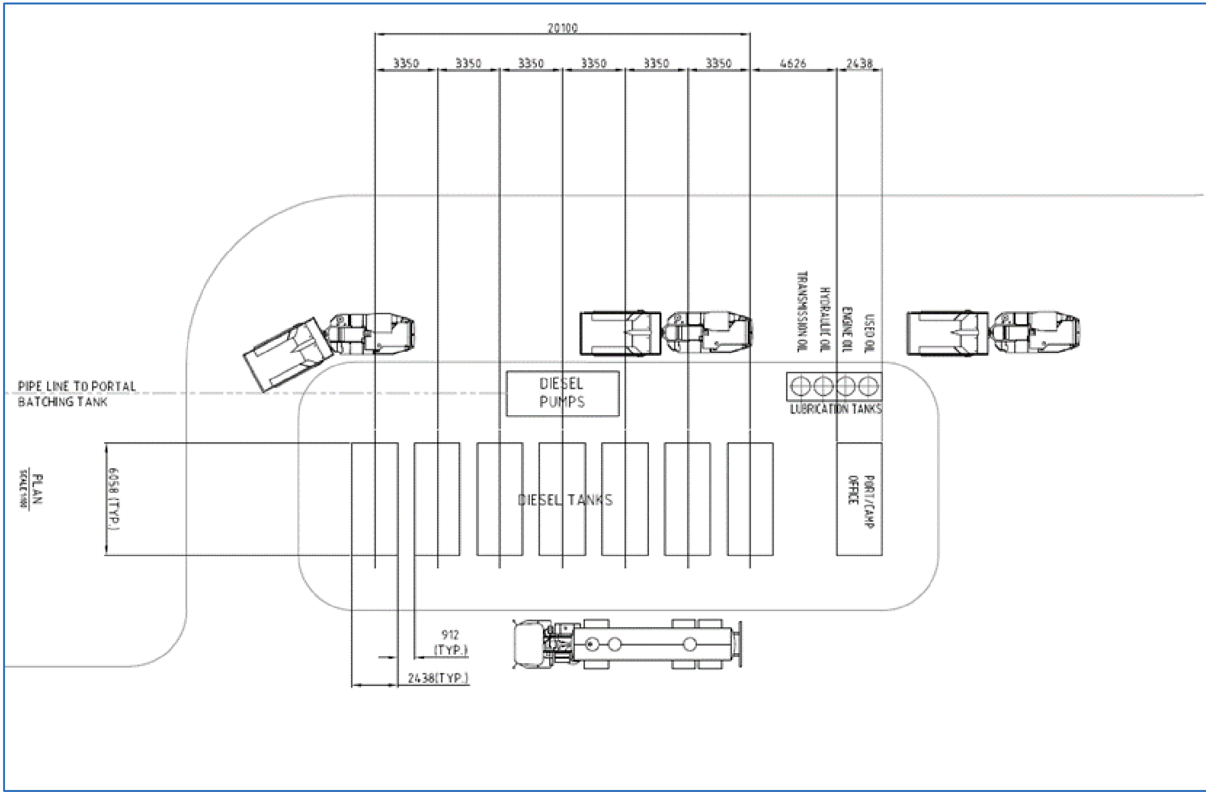
The contracted fuel supplier will deliver diesel and lubrication oils, in bulk, by road. To mitigate risks associated with road delivery, one month's supply of diesel will be stored on surface in containerised storage containers.

Tier 2 trackless mobile equipment will be purchased to operate on readily available 500 ppm sulphur-content fuel.

Surface diesel storage will be in double-walled, containerised, fuel storage tanks. Seven 72,000 L storage tanks will provide one month's storage capacity. The tanks will be interconnected and discharge through a single containerized pumping facility. The various grades of lubricants will also be stored at the fuel facility in separate lubrication tanks.

The surface diesel and lubrication storage facility layout is shown in Figure 16.17.

Figure 16.17 Surface Diesel and Lubrication Storage Facility



Diesel will be pumped from the surface storage tanks to a 28,000 L doubled-walled containerised batching tank positioned at the portal.

From the portal batching tank, diesel will be gravity fed underground by means of a 50 NB open ended, galvanized, carbon steel, flanged pipe. The pipeline will be fitted with a flow safety valve to shut off flow in the event of a rupture in the line. Two energy dissipaters, installed at equal distances between the batching tank and underground storage tank, will limit the pipeline velocity to 0.5 m/s.

In order to reduce the risk of fire in the main conveyor decline, the diesel supply pipe will be installed in the service decline.

A 28,000 L doubled-walled containerised underground diesel storage tank will be located in the underground satellite workshop. A level transmitter on the underground diesel storage tank will control the batching process. Only when the tank is empty will batching from surface take place. The batching operation will prevent overfilling of the underground tank and also ensure that the diesel pipeline is empty unless diesel is being transferred.

Lubrication oil will be transported underground in lubrication cassette trucks and will be stored in the underground workshop lubrication tanks.

16.2.7.3 Water Management and Dewatering Systems

The mine dewatering system has been designed to handle both the mining production water requirements and the fissure water ingress. Groundwater ingress has been estimated at 7 L/sec/km^2 by Golder Associates.

Mine service water (MSW) is piped underground from a MSW dam on surface. The MSW is required for drill rigs, used for various washing down activities in mining areas and the cleaning of mobile equipment at the underground workshop.

The total MSW requirement has been calculated using an industry norm of 0.5 t of water per tonne of rock mined.

The surface MSW dam will be sized to provide 36 hours capacity (2.5 ML).The permanent 300 NB feed pipe line will be installed with a booster pump initially until sufficient pressure is generated by gravity. Pressure reducing stations will be installed as the mine develops, controlling water pressure to 100 kPa.

Surface Settling System

The settling system on surface will consist of two, rectangular, HDPE-lined, earth dams. Clean water will overflow into a transfer dam and will be pumped to the MSW dam. The settling system has been sized for the permanent arrangement of the mine. Each settling dam will have a capacity of 1,000 m^3 allowing 7.7 hours of residence time for settling to take place. A settling dam will be allowed to silt up to 50% of its volume before the second dam is brought into operation. The de-commissioned dam will be allowed to dry and the silt will then be reclaimed and transported to the process plant. It has been estimated that, under normal operating conditions, each settling dam will be switched and cleaned out every 2.5 years.

Underground Dewatering System

The underground dewatering system will consist of a temporary pumping system designed to handle the MSW (including fissure water ingress) required for the development of the conveyor and service declines.

The permanent pumping system, designed to handle the MSW required at steady state, will be positioned at the top of the Centrale access decline and Kansoko Sud roadway. The system will be commissioned when mined-out SR&P areas are available to construct underground settling and clean water dams.

Temporary Pumping System

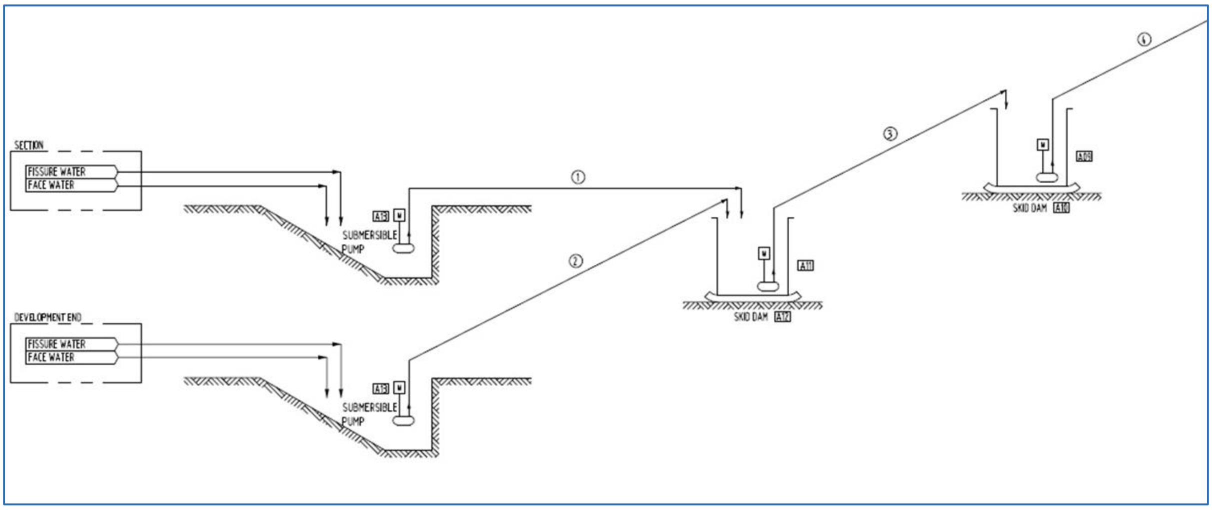
Dirty water will be pumped from the development ends and SR&P mining faces using electric submersible pumps into nearby skid-mounted tanks. The tanks will incorporate a vertical spindle pump to pump the dirty water, via a series of tanks at 55 m intervals, to the surface settling dams.

These skid-mounted tanks will later pump to permanent dams , positioned at 315 m intervals along the decline, when these dams are developed and commissioned. The dams will be equipped with vertical spindle pumps which will pump in series to the surface settling dams.

The dirty water pumping system, due to limited water storage capacity, will run continuously in order to control the ingress of fissure water.

Figure 16.18 shows dirty water handling from the development ends and mining face to the skid-mounted tanks.

Figure 16.18 Dirty Water handling from the Development Ends and Mining Faces to the Skid-mounted Tanks

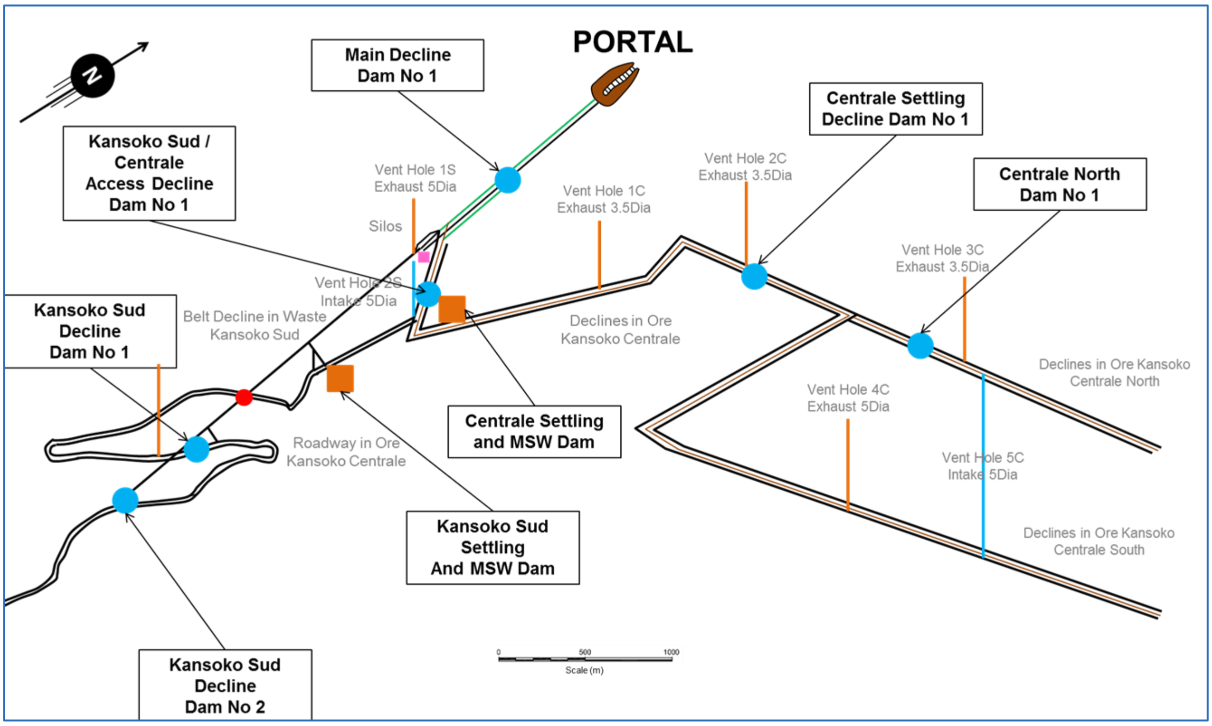


Permanent Pumping System

The permanent pumping arrangement will be constructed and commissioned once sufficient mined-out SR&P area is available to accommodate underground settling and MSW dams. The dirty water from the mining areas will be pumped to underground settling and MSW dams positioned at the top of the Centrale access decline and the Kansoko Sud roadway.

Figure 16.19 shows dam and settler locations: permanent pumping system.

Figure 16.19 Settling and MSW Dam Locations for the Permanent Pumping System



Each underground settling system will consist of two parallel settling dams (one operating, the other drying and cleaning out) that overflow into their respective MSW dams. Each settling dam will have a capacity of 800 m^3 providing 3.2 hours of residence time for settling to take place. The settling dams will be switched every four to six months. The de-commissioned dam will be allowed to dry and the silt reclaimed by a LHD and loaded onto the conveyor system.

Clear water pumps will be installed at the two underground MSW dams to pump the excess water to the Kansoko Sud/Centrale access decline dam no. 1 and main decline dam no. 1. These two dams will be equipped with multi-stage clear water pumps to pump excess water back to the surface MSW.

16.2.7.4 Fire Protection Systems

Fire suppression and fire detection systems will be installed to protect surface and underground infrastructure and equipment.

Surface

A ring main fire water reticulation system will be in place to supply fire hydrants protecting surface buildings and infrastructure. Hose reels and portable fire extinguishers will be placed in all buildings. Fire trolleys equipped with 50 kg dry powder extinguishers will be provided at the containerised diesel and oil storage tanks. Fire detection systems will be installed at all electrical substations.

Underground

Fire detection sensors and fire water sprays will be installed on all conveyors. Fire extinguishers will be placed strategically along the length of the conveyors. Smoke detectors will be installed at strategic points throughout the mine. Fire extinguishers will be installed in all electrical substations. An automatic fire water sprinkler system will be installed at the satellite workshop which will also have dry powder fire extinguishers at the refuelling bay. Fire suppression systems will be fitted to all mobile diesel-driven equipment.

16.2.7.5 Materials Handling Logistics

Material, equipment and mining stores will be delivered to the mine site warehouse by road. The warehouse will service both the process plant and mining operations. The warehouse will include a bonded store for all imported stores and equipment. The warehouse will supply the following satellite stores:

- Surface HME workshop and underground satellite workshop: Mobile production fleet spares will be kept and managed from store facilities in these workshops. Mobile fleet service exchange components will be managed from a service exchange store in the surface HME workshop. Service exchange components will be serviced at the OEM's repair and service workshop.
- Tyre workshop: Mobile production fleet tyres will be requisitioned from the warehouse and stored in a tyre store at the tyre workshop.
- Underground mining laydown areas: Mining stores will be requisitioned from the warehouse and kept in laydown areas close to underground mining operations.

16.2.7.6 Power and Communication Systems

Power for the Kamoa project is planned to be sourced from three SNEL hydro-electric power stations. Power supply to the mine will be unreliable until the three plants have been refurbished and a new double transmission line has been installed to feed Kamoa.

Medium and Low Distribution Voltages

The medium voltage distribution levels in the SNEL network in the DRC include 15 kV, 11 kV, and 6.6 kV.

Process plant distributing and reticulation on surface and underground will be at 11 kV.

Low voltage distribution at 690 V will present advantages of capital cost saving and network efficiency improvement.

Electrical Substations and Power Distribution

Power sourced from SNEL will be transmitted at 33 kV from Kolwezi to the Kamoa main consumer substation on the mine.

The mine portal substation will be supplied from the Kamoa main consumer substation located next to it. The mine portal substation will be fed by a single 33 kV feeder and will step down the voltage, using dual 20 MVA transformers, to 11 kV.

The two 20 MVA power transformers will feed indoor metal-enclosed 11 kV switchgear installed at the portal substation.

The portal substation MV switchboard will distribute power at 690 V to the following feeders: ventilation fan overhead line 1 (also feeds the Centrale paste fill plant); ventilation fan overhead line 2; refrigeration plant overhead line 1; surface mini-sub feeder 1; surface mini-sub feeder 2; portal MCC feeder 1; portal MCC feeder 2; underground feeder 1; and underground feeder 2.

Ventilation Fan Substations

The mine is to have six ventilation fan substations, each located in the vicinity of a ventilation fan. The substations will be fed by three 11 kV overhead lines from the mine portal substation. The three lines will be linked together in a ring, at the refrigeration plant substation, via the 11 kV switchboard.

The overhead line conductor has been sized to allow for redundancy. Should there be a fault on one line, the other two lines will be able to feed the six ventilation fan and the refrigeration plant substations as well as the Centrale paste fill plant.

Underground Substations

The underground distribution infrastructure will consist of four MV substations, decline 1 to 4 underground substations.

The Decline 1 underground substation will be fed, from the mine portal substation, by two 11 kV cables. The substation will accommodate an 11 kV switchboard with dual incomers and a bus-coupler. The switchboard will feed decline 2, 3, and 4 underground substations. Each decline substation will feed the mining area loads in its vicinity.

The underground 11 kV feeder cables have not been sized for redundancy. If one feeder cable is damaged, then mining production will need to be reduced until the cable has been repaired. In order to mitigate the risk, one feeder cable will be installed in the main conveyor decline and the second in the main service decline.

Overall Mine Demand

The total load, as measured on the 33 kV feeder and without power factor correction, is 35.1 MVA which excludes system cable capacitance compensation and reactive transformer losses. A 20% contingency has been added to the calculated power demand to give an assumed 38.6 MVA at a power factor of 0.85.

Emergency Generator Plant

A diesel generator will supply emergency power to critical equipment and plant in both the mining and process plant operations. Adding a 20% contingency to the calculated emergency power requirement; the emergency power requirement becomes 8.3 MVA at a power factor of 0.9.

Ventilation Fan

Power will be available to run one ventilation fan per fan district. However, sufficient air circulation may be available by manually opening the surface ventilation fan control louvres and as a result, the running of the surface fans may be scheduled to reduce emergency power demand.

Bulk Air Cooler

Emergency power will be available to run the BAC providing cool air underground for a limited period to allow the safe evacuation of personnel.

Refuge Station Compressed Air Supply

In the event of an underground fire that interrupts power supply, emergency power will be available to run the compressor station providing compressed air to the refuge stations.

Underground Pumping

Emergency power will be available for the underground pumps to pump fissure water out of the mine to prevent flooding.

Access Control

The access control system will be backed up by a UPS so that control of access to the underground mine is maintained.

Communication and Control Systems

Control and monitoring of the mine conveyor systems, water handling systems, surface ventilation fans and refrigeration plant will be critical to ensure the safe and efficient operation of the mine. The ability to monitor plant and equipment remotely is essential.

Control Room

A dedicated control room will be located on the surface. The control room will be manned 24 hours per day. The following will be monitored and controlled from the control room:

- Underground conveyor transport system.
- Water handling system.
- Main fans and refrigeration plant.
- Paste fill plant.
- Central blasting system.
- Personnel and vehicle tracking.

Voice and Data System

The mine-wide voice and data equipment will be installed in the main office server room from where a fibre network will be reticulated throughout the surface buildings. A PABX system will also be installed in this server room to facilitate surface communication. The underground communication system will incorporate a leaky feeder which will be connected to surface by means of a dedicated fibre cable.

Process Control System

The process control system will consist of a master Programmable Logic Controller (PLC) and Supervisory Control and Data Acquisition (SCADA) system configuration, housed in the mine server room on surface, which will be adjacent to the control room. The master PLC will be linked to the underground remote I/O panels (RIO panels) via a dedicated fibre backbone using an industrial Ethernet network communication protocol.

Central Blasting System

Provision has been made for a central blasting system to control and monitoring blasting. A dedicated personal computer for the blasting system, complete with the required network switches and network security, will be installed in the control room.

Conveyor and Pump Controls

Intelligent conveyor and pump protection will be employed to enable remote control and monitoring of the conveyors and pumps.

16.2.8 Mining Equipment

16.2.8.1 Mobile Equipment

All mobile equipment will be diesel driven trackless diesel equipment. Equipment requirements were estimated as dedicated fleets for: development, SR&P and D&F. A fleet of support vehicles has been included to support the development and mining process and a small fleet of surface services vehicles will be purchased for surface infrastructure maintenance. Fleet size and composition will fluctuate with demand and changes in the work requirements. The typical fleet requirements for each activity are shown in Table 16.12 to Table 16.14.

Table 16.12 Development Fleet

Unit	Total
LHD (17 t)	6
Twin Boom Drill Jumbo	4
Dump trucks (51 t)	4
Rockbolter	4
Cable Bolter	2
Explosive charger	4
Personnel transporter (28 seater)	2
Utility vehicle – 4WD	3
Utility vehicle – explosives	2
Scissor lift	2
Grader	2
Shotcrete sprayers	2
Total	**37**

Table 16.13 SR&P Fleet

Unit	Total
LHD (17 t)	2
Twin Boom Drill Jumbo	2
Dump trucks (51 t)	4
Rockbolter	2
Personnel transporter (28 seater)	2
Explosive charger	2
Utility vehicle – 4WD	2
Utility vehicle - explosives	1
Total	**17**

Table 16.14 D&F Fleet

Unit	Per Crew
LHD (17 t)	3
Twin Boom Drill Jumbo	4
Rockbolter	4
Personnel transporter (28 seater)	2
Explosive charger	3
Utility vehicle – 4WD	2
Utility vehicle - explosives	1
Total	19

16.2.8.2 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.

Water Handling Pumps

- Temporary Pumping

Electric submersible pumps will pump dirty water from the mining face to the skid-mounted steel tanks; 10 kW vertical spindle pumps will pump water from the skid-mounted steel tanks to the decline dams. The 45 kW vertical spindle pumps will pump, in series, from the decline dams to surface.

- Permanent Pumping

Warman C5 110 kW dirty water pumps will pump water from the permanent decline dams to the underground settling dams.

Multistage pumps will pump clear water from the underground MSW dams to the main decline dam and from this main decline dam to surface.

Decline Conveyors

- Main Decline Conveyor

The steel structure for the main decline conveyor will be suspended from the hangingwall and supported from the footwall at conveyor drives and ore loading points. The 1,050 mm wide conveyor system will be fitted with three 250 kW drive units, two primaries and one secondary. A class 2500 steel cord belt will be installed.

- Decline Conveyors

The steel structures for the decline conveyors will also be suspended from the hangingwall and supported from the footwall at conveyor drives and ore loading points. Each 1,050 mm wide decline conveyor system will be fitted with two 250 kW drive units, one primary and one secondary. Class 2000 fabric belting will be installed.

- Rock Beakers

Electro-hydraulic rock breakers will be fitted to all ore loading stations on the section conveyors, ore passes and on top of the ore storage silos. The rock breakers will be pedestal mounted with 170° swing arm rotation and fitted with excavation-style booms providing increased depth of reach.

Main Surface Ventilation Fans

All fan electrical control gear will be housed in a suitable building. The fan drive motors will be weatherproof and will not require protection from the elements.

- Single Fan Drift

A single fan drift will ventilate raise 1C. The centrifugal fan will be driven by a 550 kW, 11 kV motor directly coupled to the fan impeller. The fan ducting will be fitted with non-return doors.

- Bifurcated Fan Drift

The bifurcated fan drift will ventilate raises 2C, 3C, and 3S. Each centrifugal fan will be driven by a 550 kW, 11 kV motor directly coupled to the fan impeller. The two fan ducts will each be fitted with non-return doors.

- Trifurcated Fan Drift

The trifurcated fan drift will ventilate raises 1S and 4C. The fan installation will be identical to the single and bifurcated fan installations.

- Underground Satellite Workshop

A 2 m diameter ventilation raise equipped with a 75 kW, 690 V axial flow fan, on surface, which will provide ventilation to the underground satellite workshop.

Surface Refrigeration Plant and Bulk Air Cooler

- Refrigeration Plant

The refrigeration plant installed on surface will comprise a 7 MW standard R134a centrifugal compressor machine, with shell and tube heat exchangers. The benefits of installing two 3.5 MW machines require further investigation.

- Condenser Cooling Tower

The condenser cooling tower will be a mechanical draft, packed, counter-flow type tower. The tower will be constructed in reinforced concrete on top of a concrete water reservoir.

- Bulk Air Cooler

The BAC will be a horizontal spray chamber constructed in concrete and equipped with three 90 kW forced draught fans, 690 V motors.

- Cooling Tower

The cooling tower will be a mechanical draft, packed, counter flow concrete tower. Make-up water for the cooling tower will be sourced from a surface borehole to be drilled at the cooling tower site.

Overhead Cranes in the HME Workshop

Two electric, double girder, overhead, travelling cranes will be installed in the HME workshop. The cranes, one 10 t and one 20 t, will be equipped with maintenance platforms and overload limiters.

Wash Bay

The surface wash bay will be manually operated and equipped with high pressure water pumps and spray lances.

Tyre Press in the Tyre Workshop

An Essential Power-press 2451, designed to handle 12.00R24-50/65-1 wheel assemblies and complete with tyre carrier table, will be installed in the tyre workshop.

Engine Anti-freeze Generating Plant

An anti-freeze generating plant, complete with three chemical storage tanks, water storage, blending tank and water softener plant, will be installed adjacent to the HME workshop.

Surface Diesel and Lubrication Storage and Dispensing

- Diesel

The seven 72,000 L containerised, safe fill, diesel storage tanks providing 500,000 L of storage will be installed on surface. A modular pump house will dispense diesel to a 28,000 L containerised batching tank at the portal to be transferred underground.

- Lubrication Oil

Storage and dispensing tanks for engine (10,000 L), hydraulic (6,500 L) and transmission (6,500 L) oils, including one tank for used oil, will be installed on surface together with the diesel storage tanks.

- Grease Bins

Skid-mounted RTM 1450 grease bins with dispensing trolleys will be provided, one in the surface HME workshop and one in the underground satellite workshop. The bins will be swapped out as the grease is depleted. The grease bins are coupled to the dispensing trolley by a quick-coupling flexible hose to facilitate quick and easy swapping over of the bins.

Storage Tanks for Emulsion Explosives

Three 30 t emulsion tanks and one 2,500 L gassing solution tank will be installed on surface. Emulsion and gassing solution will be delivered by road and decanted into the tanks.

Air Compressors

The two 795 CFM air compressors to supply air to underground refuge stations and workshop pneumatic tools will be installed in a compressor house on surface.

Emergency Generator

One diesel-driven 33 kV emergency generator, adequately sized to supply emergency power to the process plant and mining operations, will be installed adjacent to the mine consumer substation.

16.2.8.3 Personnel

The labour complement was developed based on working cycle of a full calendar operation of 12 hour shifts, 365 days a year and rosters of four days on and four days off.

Mining crew complements were compiled based on the number of personnel required to operate the mobile production fleet, including support crews.

The engineering crew to service and maintain the mobile production fleet, in particular the diesel mechanic/fitter complement, was compiled based on OEM and industry standards (three machines per diesel mechanic/fitter). The composition and complement of the engineering crew was compiled based on a similar sized underground mine in Botswana and benchmarked against a similar underground mining operation in the DRC.

During the initial life of the mine approximately 40% of the personnel employed will be expatriates. The expatriate complement will be gradually phased out over a period of 12 years being replaced with personnel sourced from the Kolwezi area and thereafter from the DRC as a whole. Expatriate labour was identified based on critical job categories.

Adequate training of the local labour will be scheduled to ensure timeous replacement of the expatriate labour.

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 labor 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 labor will require training to be conducted in French and Swahili.

Pre-production training of the Owner's development and mining crews has been scheduled as follows:

Development Crew

- Training of the Owner's development crew has been scheduled for a period of six months.
- The operator training will initially be done by the OEM until the mine's instructors have received the required training and been found to be competent ("train the trainer").
- Qualified engineering personnel will be recruited and will therefore only require equipment specific training.
- The owner's development crew, after completing their training, will work alongside the contract development team until the Owner's team starts development.

Stepped Room and Pillar Mining

- Training of the first SR&P production crew of 65 will occur in two phases, 6 months and 3 months prior to the commencement of SR&P mining.
- Training of a further crew of 64 will again occur in two phases, 6 months and 3 months prior to the commencement of full SR&P production.

A fully-trained complement of 129 will then be available for SR&P production.

Drift and Fill Mining

- Training of the first D&F production crew of 81 will occur in two phases, 6 months and 3 months prior to the commencement of D&F mining.
- Training of a further crew of 84 will again occur in two phases, 6 months and 3 months prior to the commencement of full D&F production.

A fully trained complement of 165 will then be available for D&F production.

16.3 Development and Construction Schedule

The project schedule includes three major components: Development, Construction, and Production. All development and stoping designs were generated using Mine2-4D design software. A production and development schedule was produced using Enhanced Production Scheduler (EPS).

Development will be carried out by a contractor's team until first ore production, at which point the Owner's team will take over and complete the development. A training program will commence with the initiation of the decline development, included in this training will be "on the job" training underground by the experienced contractor's team.

The key development milestone dates are shown in Table 16.15.

Table 16.15 Summary of Mining Project Milestones

Milestone	Date
Decline development commences	Year 1 Month 1
Declines at silo position	Year 1 Month 12
Ore silos completed	Year 2 Month 12
Main conveyor to surface commissioned	Year 3 Month 1
Kansoko Sud decline to SR&P mining area	Year 3 Month 9
Kansoko Sud conveyor installed	Year 5 Month 2
Declines to Centrale start	Year 2 Month 5
First ore tonnes	Year 2 Month 1
First production	Year 3 Month 1

Steady state production	Year 7 Month 12

A development heading advance rate of 130 m per month was used for planning based on simulations developed to model various advance scenarios, heading configurations, and crew situations. The mining cycle schedule, derived from first principles, was completed for various mining systems and face arrangements, i.e. single heading, twin developments and triple developments.

Vertical advance rates have been planned at 55 m per month which includes piloting. This is a conservative rate and is based on the ability to remove the raisebore cuttings without interfering with parallel operations.

For the purpose of the schedule, the development metres have been broken up into:

- Lateral development metres, which include the service decline and conveyor decline development, and,
- Vertical development metres, which include all ventilation, ore pass and access development.

Development in the ore body has been classified as development in ore which includes the triple declines developed into the Centrale mining area and the Kansoko Sud roadway development. The development quantities are shown in Table 16.16.

Table 16.16 Development Quantities

Development	**Metres**
Lateral Development	7,672
Vertical Development	4,064
Total	11,736
Development in Ore	**Metres**
Lateral Development	46,816

A number of vertical ventilation raise boreholes have been planned; the raise boreholes function, diameter and length are listed in Table 16.17.

Table 16.17 Vertical Ventilation Raise Boreholes

Name	Up-cast/Down-cast	Diameter (m)	Length (m)	Timing
1S	Up-cast	5.0	217	Year 2 Month 3
2S	Down-cast	5.0	280	Year 2 Month 7
3S	Up-cast	3.5	338	Year 4 Month 4
1C	Up-cast	3.5	361	Year 3 Month 5
2C	Up-cast	3.5	362	Year 3 Month 11
3C	Up-cast	3.5	523	Year 21 Month 2
4C	Up-cast	5.0	742	Year 5 Month 12
5C	Down-cast (bulk air cooler)	5.0	825	Year 6 Month 7
Satellite workshop	Up-cast	2.0	280	Year 2 Month 5

Construction

Development of the construction schedule involved listing the required mine infrastructure and facilities (surface and underground), followed by estimating the construction and erection durations.

The following assumptions were made when compiling the schedule:

- The development of both access declines from Kansoko Sud portal would be concurrent.
- The development of the access declines would start in the mid-2016.
- Design, procurement and delivery activities have not been specifically scheduled. However, sufficient time has been allowed for these activities to be undertaken.

The construction of the main decline conveyor will effectively reduce mine access to a single decline. The use of a traffic light system, passing bays and a transportation schedule, will allow access to development and silo construction to continue through the service decline only.

16.4 Mine Production Schedules

The mine production schedule is summarised in Table 16.18. The annual development and mine production are shown graphically in Figure 16.20, Figure 16.21 and Figure 16.22. The annual production schedule is shown in Table 16.19. The development schedule is shown in Table 16.20.

Table 16.18 Mine Production Summary

Description	Unit	Total
Waste Development	(km)	18.9
Vertical Development	(km)	4.1
SRP Development	(km)	0.7
SRP Stoping	(km)	189.6
D&F Development	(km)	74.4
D&F Stoping	(km)	697.0
Ore Development	(km)	39.2
Total Development	(km)	322.9
Room & Pillar Ore Mined	(Mt)	25.4
	(% Cu)	3.61
Cut & Fill Ore Mined	(Mt)	43.1
	(% Cu)	4.04
Ore Development Ore Mined	(Mt)	3.4
	(% Cu)	3.40
Total Ore Mined	**(Mt)**	**71.9**
	(% Cu)	**3.86**

Figure 16.20 Kamoa 2016 PFS Development

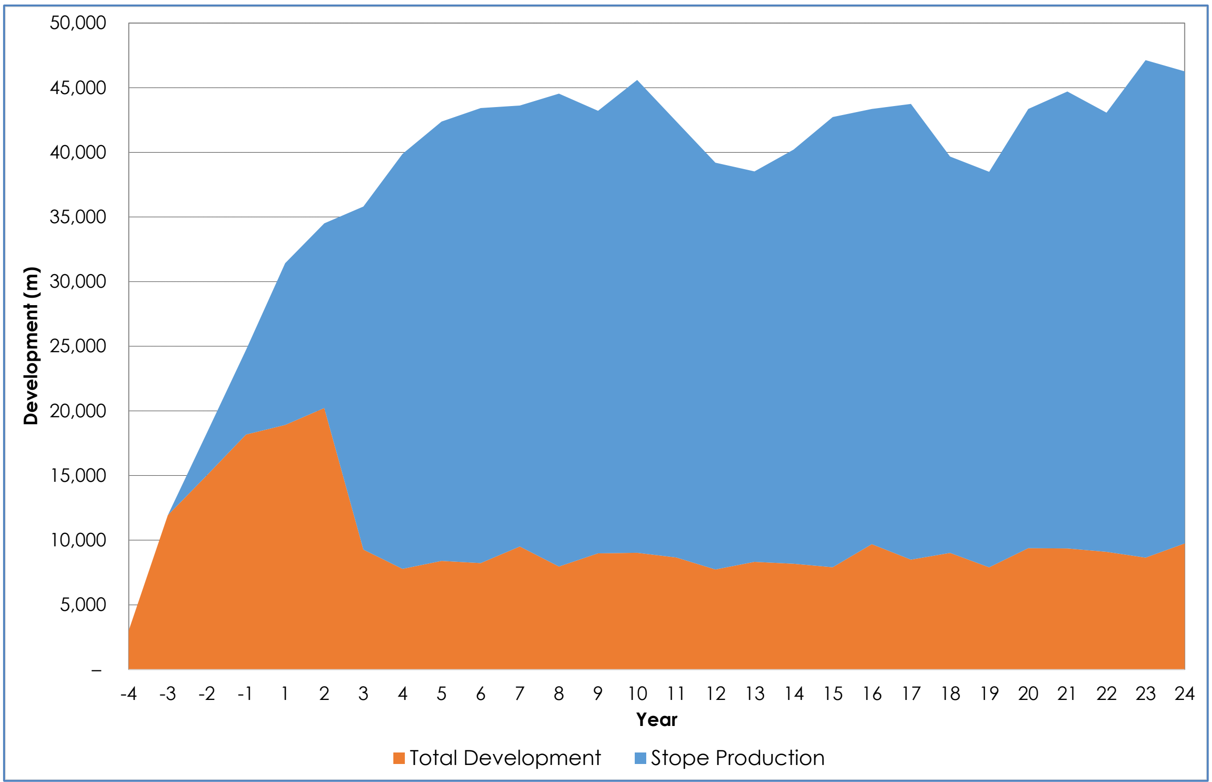


Figure by OreWin 2016

Figure 16.21 Kamoa 2016 PFS Mining Production by Source

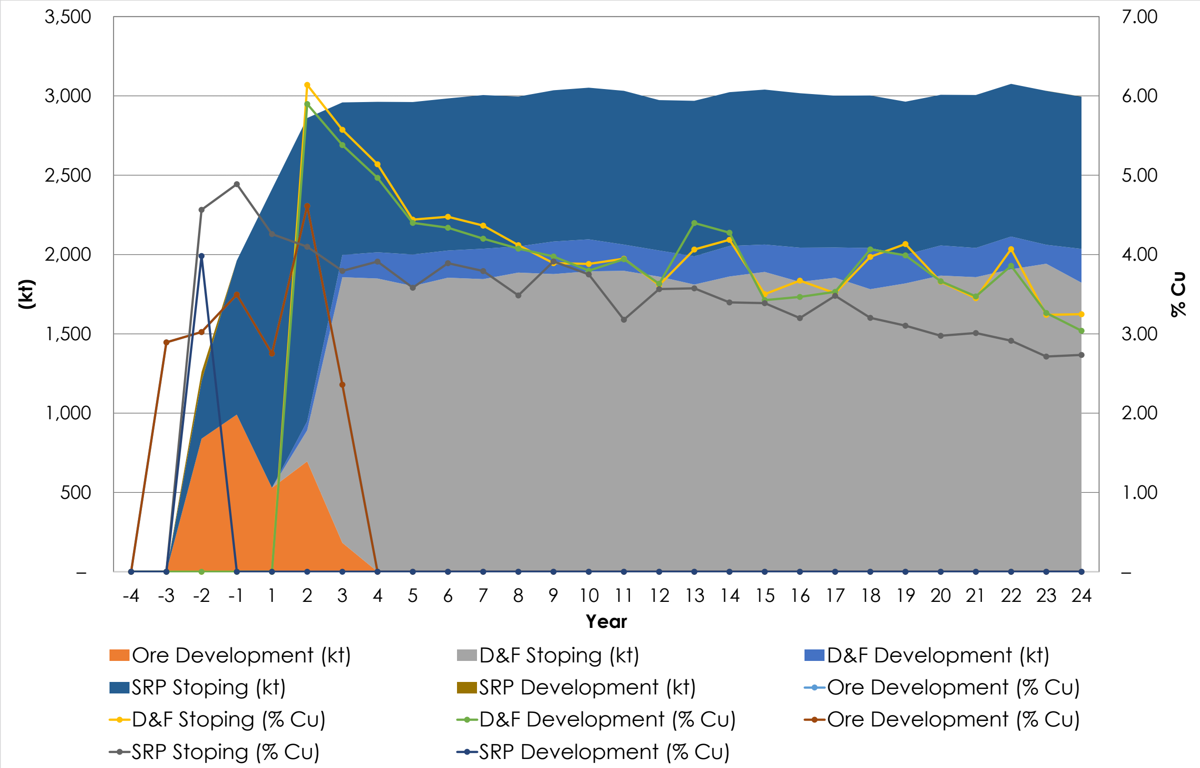


Figure by OreWin 2016.

Figure 16.22 Kamoa 2016 PFS Mining Production

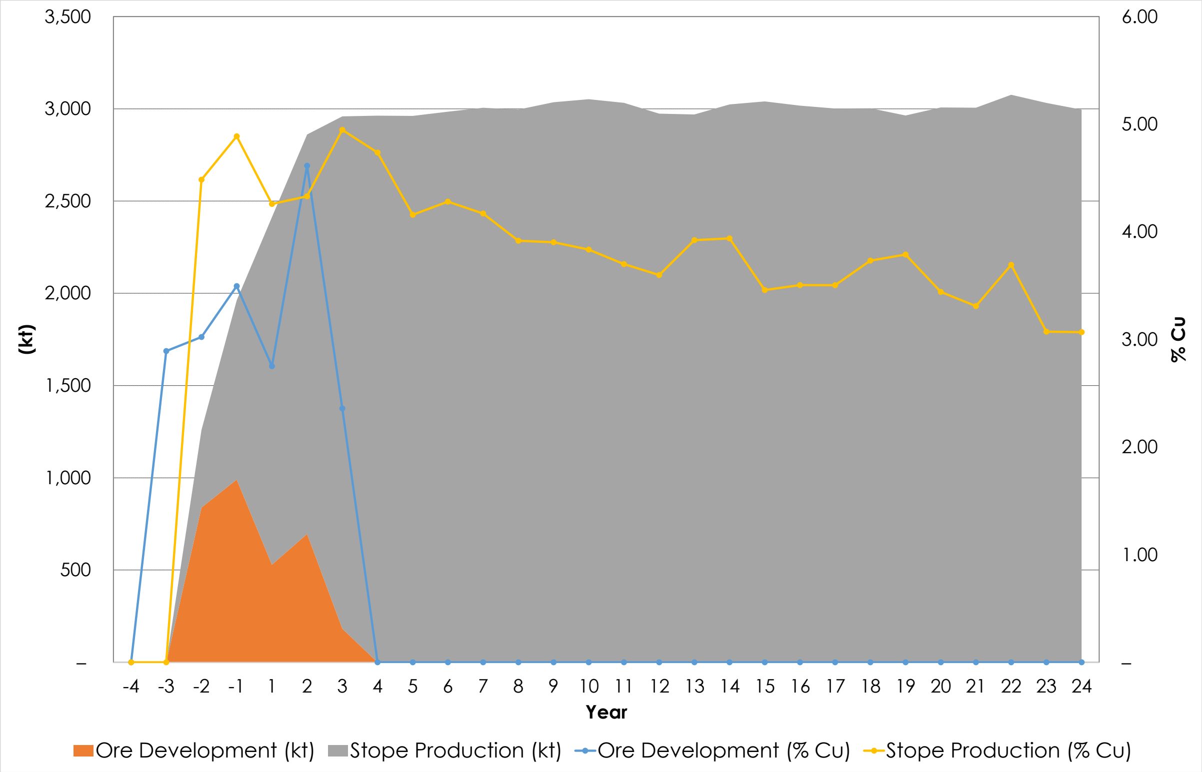


Figure by OreWin 2016.

Table 16.19 Mine Production Schedule

Description	Unit	Total	Project Time (Years)													
			-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11
Room & Pillar Ore Mined	(kt)	25,445	–	–	421	972	1,882	1,909	973	949	966	969	967	945	951	963
	(% Cu)	3.61	–	–	4.48	4.89	4.26	4.10	3.78	3.94	3.59	3.90	3.78	3.49	3.91	3.76
Cut & Fill Ore Mined	(kt)	43,060	–	–	–	–	–	281	1,853	2,042	2,027	2,032	2,042	2,062	2,079	2,069
	(% Cu)	4.04	–	–	–	–	–	6.01	5.54	5.11	4.44	4.42	4.38	4.09	3.87	3.90
Ore Development Ore Mined	(kt)	3,387	–	151	839	991	529	696	182	–	–	–	–	–	–	–
	(% Cu)	3.40	–	2.89	3.02	3.50	2.75	4.61	2.36	–	–	–	–	–	–	–
Total Ore Mined	**(kt)**	**71,893**	**–**	**151**	**1,260**	**1,963**	**2,411**	**2,885**	**3,009**	**2,990**	**2,993**	**3,001**	**3,008**	**3,007**	**3,030**	**3,032**
	(% Cu)	**3.86**	**–**	**2.89**	**3.51**	**4.18**	**3.93**	**4.41**	**4.78**	**4.74**	**4.16**	**4.25**	**4.19**	**3.90**	**3.88**	**3.85**

Description	Unit	Total	Project Time (Years)													
			12	13	14	15	16	17	18	19	20	21	22	23	24	25
Room & Pillar Ore Mined	(kt)	25,445	964	948	970	973	974	979	954	951	978	972	973	973	970	–
	(% Cu)	3.61	3.16	3.57	3.57	3.40	3.39	3.20	3.49	3.20	3.12	2.97	3.00	2.89	2.70	–
Cut & Fill Ore Mined	(kt)	43,060	2,065	2,064	2,033	2,041	2,041	2,031	2,048	2,047	2,025	2,046	2,029	2,058	2,045	–
	(% Cu)	4.04	3.91	3.62	4.12	4.19	3.48	3.65	3.51	3.97	4.12	3.65	3.43	4.08	3.20	–
Ore Development Ore Mined	(kt)	3,387	–	–	–	–	–	–	–	–	–	–	–	–	–	–
	(% Cu)	3.40	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Total Ore Mined	**(kt)**	**71,893**	**3,028**	**3,012**	**3,004**	**3,014**	**3,015**	**3,011**	**3,002**	**2,998**	**3,003**	**3,019**	**3,002**	**3,030**	**3,015**	**–**
	(% Cu)	**3.86**	**3.67**	**3.60**	**3.95**	**3.94**	**3.45**	**3.50**	**3.50**	**3.73**	**3.79**	**3.43**	**3.29**	**3.70**	**3.04**	**–**

Table 16.20 Mine Development Schedule

Item	Units	Total	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10
Waste Development	(km)	18.9	3.0	10.1	2.6	1.1	0.7	0.9	–	–	–	–	–	–	–	–
Vertical Development	(km)	4.1	–	0.8	0.8	0.4	0.7	0.9	–	–	–	–	–	–	–	–
SRP Development	(km)	0.7	–	–	0.7	–	–	–	–	–	–	–	–	–	–	–
SRP Stoping	(km)	189.6	–	–	2.5	6.5	12.5	11.3	5.6	5.4	5.5	5.8	6.9	5.5	6.2	6.2
D&F Development	(km)	74.4	–	–	–	–	–	0.7	1.6	2.3	2.9	2.5	2.6	2.4	2.8	2.8
D&F Stoping	(km)	697.0	–	–	–	–	–	2.3	19.3	24.3	25.6	27.0	24.6	28.6	25.2	27.5
Ore Development	(km)	39.2	–	1.8	9.1	10.6	5.7	7.4	2.1	–	–	–	–	–	–	–
Total Development	**(km)**	**322.9**	**3.0**	**11.9**	**15.0**	**18.2**	**18.9**	**20.2**	**9.3**	**7.8**	**8.4**	**8.2**	**9.5**	**8.0**	**9.0**	**9.0**

Item	Units	Total	11	12	13	14	15	16	17	18	19	20	21	22	23	24
Waste Development	(km)	18.9	–	–	–	–	–	0.5	–	–	–	–	–	–	–	–
Vertical Development	(km)	4.1	–	–	–	–	–	0.5	–	–	–	–	–	–	–	–
SRP Development	(km)	0.7	–	–	–	–	–	–	–	–	–	–	–	–	–	–
SRP Stoping	(km)	189.6	6.4	5.6	6.1	5.7	5.4	6.2	6.0	5.8	5.5	6.7	6.8	6.2	6.9	6.5
D&F Development	(km)	74.4	2.2	2.1	2.2	2.4	2.5	3.0	2.5	3.2	2.4	2.7	2.6	2.9	1.8	3.3
D&F Stoping	(km)	697.0	25.1	23.7	21.9	23.9	26.9	24.5	26.8	21.6	22.7	24.6	26.0	24.9	29.8	26.8
Ore Development	(km)	39.2	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Total Development	**(km)**	**322.9**	**8.7**	**7.7**	**8.3**	**8.2**	**7.9**	**9.7**	**8.5**	**9.0**	**7.9**	**9.4**	**9.4**	**9.1**	**8.6**	**9.7**

16.5 Alternative Mining Method Controlled Convergence Room and Pillar

In parallel with the study an alternative mining method was investigated for its suitability for use on the Kamoa deposit. The method is controlled convergence room and pillar and does not use fill but instead pillars are stripped to allow the controlled convergence of the backs and floors. This would provide significant cost savings as fill is not required and the extraction ratios are increased. In this controlled convergence room and pillar method the primary pillars are designed to fail and remain in an elastic state in the post-failure phase. The primary pillars, depending on the degree of their disintegration, are mined to form residual pillars that are left in the mined-out workings.

A preliminary study of Kamoa by KGHM Cuprum R&D Centre Ltd. (KGHM Cuprum) suggests that the method would be applicable. Controlled convergence room and pillar is used by KGHM Polska Miedź S.A. (KGHM) at its Legnica-Glogów Copper Belt operations in Poland. Mining at the KGHM mines has occurred continuously since the 1960's. Since the mid 1990's KGHM has conducted operations using the controlled convergence room and pillar system to mine ore deposits up to 7 m thick and to a depth up to 1,200 m. KGHM Cuprum technical personnel visited Kamoa site and hosted meetings with the Kamoa study team as part of the KGHM Cuprum study. The KGHM Lubin and Rudna mines were visited by Bernard Peters, QP, and included an extensive review of the underground operations and meetings with KGHM Cuprum technical personnel.

The principal mining operations of KGHM comprise three large underground mines which extend over a strike length of approximately 40 km. These three underground mines operate at depths of between 600 and 1,250 m below surface and the combined annual production totals approximately 30 million tonnes of ore per year.

Prior to the extraction of a mining section the ore zone is contoured by preparatory workings, usually two development headings, to verify geological continuity and ore grade.

After delineating a mining section, the primary extraction phase involves developing parallel rooms and cross-cuts that result in the creation of a series of rectangular primary pillars which support the rock mass above the working area.

These primary pillars are designed to fail and depending on the degree of disintegration, they are mined in a secondary extraction phase to form residual pillars that are left in the mined-out zone where they act as supports mitigating the bending of the roof strata above the mining section.

With the completion of the primary and secondary mining phases the controlled convergence room and pillar method is capable of extracting 75% to over 90% of the in-situ ore.

Benefits of the controlled convergence room and pillar stoping include:

- Controlled convergence room and pillar stoping not limited by a depth of 600 m,
- No increase in geometry of primary pillars with depth of mining,
- Progressive recovery of ore from the primary pillars as the residual pillars are formed,
- Progressive recovery of ore from the preparatory workings protection pillars,

- No backfill required when mining ore deposits up to 7 m thick,
- The smooth subsidence of strata overlaying the working area and in the mined-out zone as pillars fail under the load from the destressed and delaminated rock mass,
- Increased extraction ratios due to primary pillar geometry, recovery of ore when forming residual pillars and the progressive recovery of the protection pillars.

The initial analyses of Kamoa geological deposition conditions and the geotechnical properties of the deposit rocks and roof rocks indicated the suitability of the controlled convergence room and pillar method and the favourable conditions for mining.

The significantly improved extraction ratios delivered by the controlled convergence room and pillar method when compared to extraction ratios for the SR&P and D&F mining methods indicates the need to further assess the merits of adopting this mining method for the Kamoa deposit. As a sensitivity the Kamoa 2016 PFS economic analysis was modified to remove the capital and operating costs for paste fill, the results of this sensitivity are shown in Table 16.21 and Table 16.22.

Table 16.21 Mining Method Comparison - Overall Results

Item	Unit	SR&P and D&F (With Fill)	Controlled Convergence Room and Pillar (No Fill)
Ore Processed			
Quantity Ore Treated	kt	71,893	71,893
Copper Feed Grade	%	3.86	3.86
Concentrate Produced			
Copper Concentrate Produced	kt (dry)	6,106	6,106
Copper Recovery	%	86.36	86.36
Copper Concentrate Grade	%	39.20	39.20
Contained Cu in Concentrate	Mlb	5,277	5,277
Key Financial Results			
Initial Capital	US$M	1,213	1,155
Mine Site Cash Cost	US$/lb Payable Cu	0.75	0.61
Total Cash Costs	US$/lb Payable Cu	1.48	1.35
Site Operating Costs	US$/t ore	53.22	43.54
After Tax $NPV_{8\%}$	US$M	986	1,182
After Tax IRR	%	17.2%	18.9%
Project Payback Period	Years	4.6	4.3

Table 16.22 Room and Pillar Mining Method Comparison Financial Results

Comparative After Tax NPV ($M)		
	SR&P and D&F (With Fill)	**Controlled Convergence Room and Pillar (No Fill)**
Discount Rate	**PFS Base Case**	**PFS Sensitivity**
Undiscounted	4,096	4,631
4.0%	2,036	2,344
6.0%	1,429	1,672
8.0%	986	1,182
10.0%	657	819
12.0%	409	546
Internal Rate of Return	17.2%	18.9%
Project Payback Period (Years)	4.6	4.3

KGHM Operations

Ivanhoe engaged KGHM Cuprum to assess the suitability of adopting the controlled convergence room and pillar mining system used by the KGHM Legnica-Glogów Copper Belt operations at the Kamoa Project.

The findings by KGHM Cuprum are detailed in the report titled "Assessment and verification of extraction ratios for room and pillar mining for KAMOA Project based on experience of Legnica-Glogów Copper district", October 2015.

KGHM is a major mining, smelting and refining company and is one of the world's leading producers of electrolytic copper and refined silver.

The copper-silver deposit in the Legnica-Glogów Copper Belt area is one of the largest polymetallic deposits in the world. The Legnica-Głogów Copper Region or Legnicko-Głogowski Okręg Miedziowy (LGOM) covers an area of over 750 km², in the south–west of Poland.

Mining of copper ore from the mines operating in the LGOM area has occurred continuously since the 1960's. During the 50 years of activity over a billion tonnes of ore and 18 million tonnes of copper has been produced.

KGHM's principal mining operations in the LGOM area comprise three large underground mines, Lubin, Polkowice-Sieroszowice and Rudna, which extend over a strike length of approximately 40 km. These three underground mines operate at depths of between 600 and 1,250 m below surface and the combined annual production totals approximately 30 million tonnes of ore per year.

KGHM employs the controlled convergence room and pillar mining method at all three mines. Variations of the basic system are adapted to suit the changing local conditions within the deposit, depending on ore thickness and the geotechnical parameters of the orebody and the surrounding rocks.

The presence of fault zones within the LGOM ore deposit determines the geometry of the larger production regions. These production regions are divided into mining sections, where the dimensions are generally determined by:

- Continuity of deposit mineralisation,
- Assumed production rates and ore grades,
- Ventilation constraints,
- Development of a relatively long mining front, maximising the number of mining faces to provide operation flexibility and continuity of production in case of geological or geotechnical issues.

Major pillars are used to protect shafts, permanent underground installations and travelways from the impact of the mining activity in the mining section. These zones are excluded from extraction until they have performed their final ventilation and transport functions.

To verify geological continuity and ore grade prior to the extraction of a mining section, the ore zone is contoured by preparatory workings, usually two development headings.

Each set of preparatory workings is used for two adjacent mining sections. Protection pillars separate the preparatory workings from the mining activity in the mining sections. These protection pillars are successively extracted as the mining front within the mining section progresses.

After delineating a mining section, the primary extraction phase involves developing parallel rooms and cross-cuts that result in the creation of a series of rectangular primary pillars which support the rock mass above the working area.

These primary pillars are designed to fail, described by KGHM Cuprum as being in the post-failure phase. Depending on the degree of disintegration, ore is removed from the primary pillars to reduce their size in a secondary extraction phase to form residual pillars that are left in the mined-out zone. These final residual pillars act to mitigate the convergence of the roof strata ensuring its slow convergence above the working area and in the mined-out workings.

A typical layout of the controlled convergence room and pillar method is shown in Figure 16.23.

Figure 16.23 Controlled Convergence Room and Pillar layout

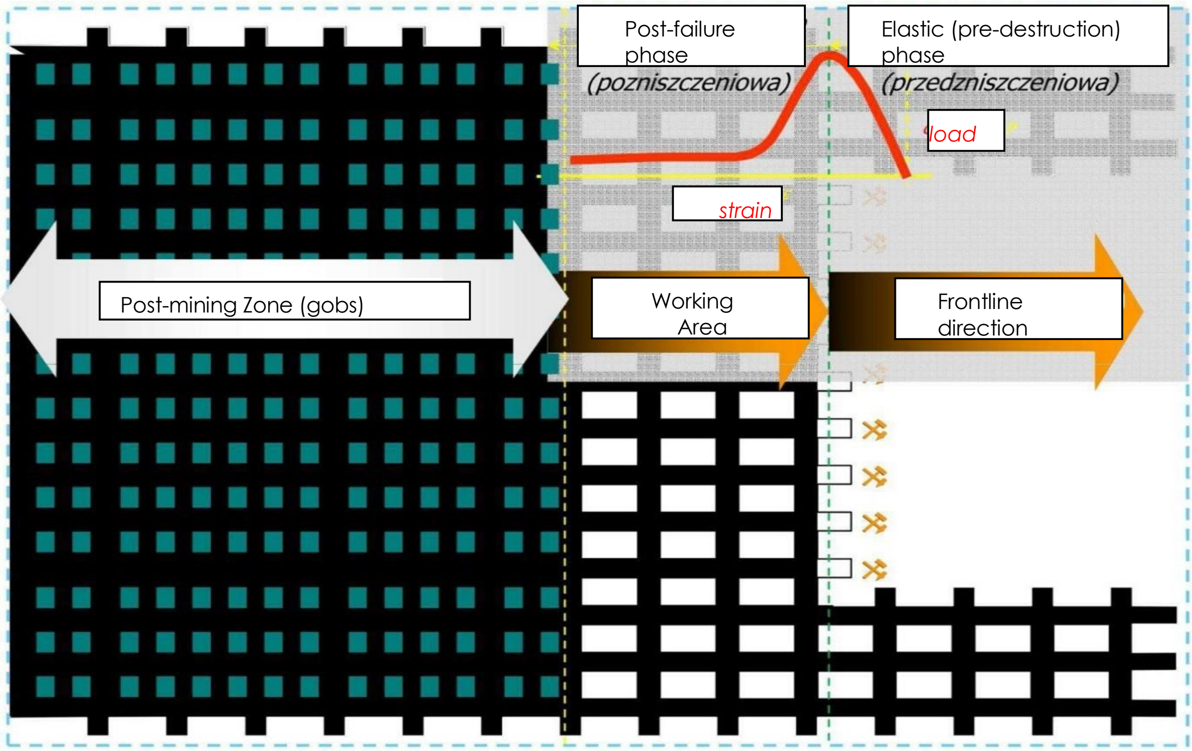
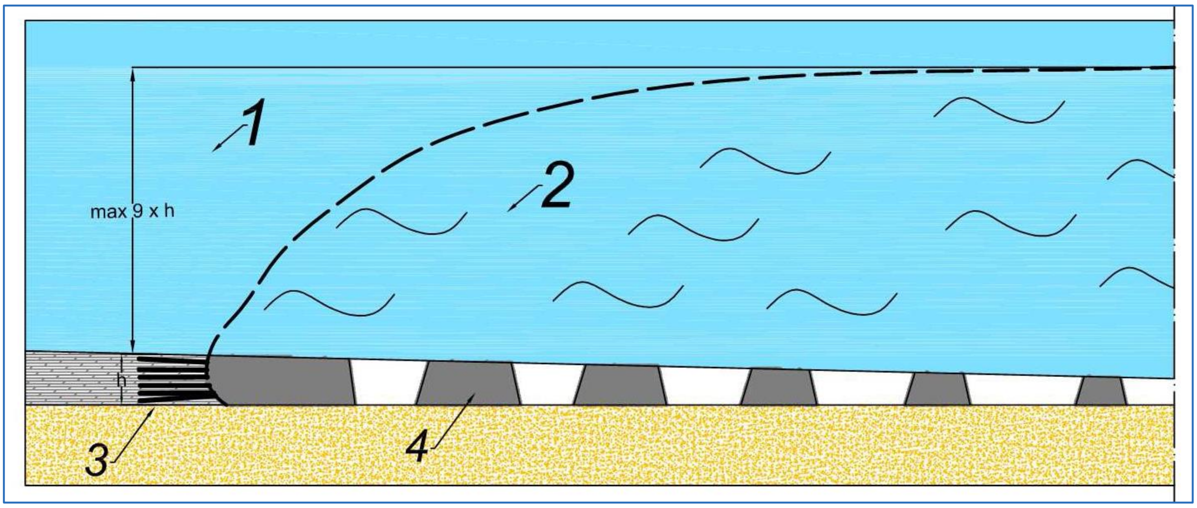


Figure 16.24 illustrates the extent of the distressed and delaminated rock mass above the working area and the behaviour of the pillars in the post-mining zone.

Figure 16.24 Outreach of the Destressed Rock Mass Area in the Controlled Convergence Room and Pillar System



1 – rock mass prior to extraction; 2 – destressed and delaminated rock mass; 3 - blasting holes; 4 - primary pillars

Since the mid 1990's operations have been conducted using the controlled convergence room and pillar method to mine ore deposits up to 7 m thick and to a depth up to 1,200 m. Variations to the system are adapted to suit local geological and mining conditions, including mining thicker ore zones and instances of increasing dip up to 35°.

With the completion of the primary and secondary mining phases the controlled convergence room and pillar method is capable of extracting 75% to over 90% of the in-situ ore.

When mining thicker portions of the ore zone, greater than 7 m, and when mining under areas requiring protection of surface structures backfill is used.

Benefits of the controlled convergence room and pillar method include:

- Controlled convergence room and pillar method not limited by 600 m depth,
- No increase in geometry of primary pillars with depth of mining,
- Progressive recovery of ore from the primary pillars as the residual pillars are formed,
- Progressive recovery of ore from the preparatory workings protection pillars,
- No backfill required when mining ore deposits up to 7 m thick,
- The smooth subsidence of strata overlaying the working area and in the mined-out zone as pillars fail under the load from the destressed and delaminated rock mass,
- Increased extraction ratios due to primary pillar geometry, recovery of ore when forming residual pillars and the progressive recovery of the protection pillars.

Suitability of Controlled Convergence Room and Pillar System to the Kamoa Deposit

KGHM Cuprum analysed the current geological, geotechnical, hydrogeological and tectonic exposure data to determine the suitability of mining the Kamoa deposit using the controlled convergence room and pillar method.

KGHM Cuprum also carried out additional petrographic testing of the rocks in the mining profile of the Kamoa deposit. The analyses indicated favourable conditions for mining using the controlled convergence room and pillar mining method at the Kamoa deposit.

Controlled Convergence Room And Pillar Assumptions for Kamoa

The mining planned for the first phase of the Kamoa deposit covers the deposit up to 6 m thick, dipping up to 12.5° and at depths to 600 m.

KGHM Cuprum noted that the adoption of the controlled convergence room and pillar method assumes that surface subsidence of the Kamoa terrain will occur. KGHM Cuprum suggested that the existing surface constraints indicate that mining without specific protection against excessive deformation is possible and in practise the use of long, relatively straight-line mining fronts helps in minimising the impact of subsidence.

The proposed dimensions for the primary pillars are detailed below:

Deposit thickness (metres):	4	6
Width (metres):	6	8
Length (metres):	7–8	9–10

The width of the extraction panel (the active mining front) was determined at approximately 300 m, with the extraction panel length being unlimited (limited only by faulting of the deposit and ventilation constraints).

The width of development used to create the rooms and cross-cuts, during the panelling phase is up to 7 m wide.

Where the deposit dip is greater than 12.5° the controlled convergence room and pillar system would be varied to include a combination of rooms and cross-cuts mined diagonally against the dip of the ore body to generate an apparent dip that will allow the use of mechanised equipment.

This method of accessing steeper dipping deposits with mechanised equipment is conceptually similar to that used in the SR&P mining method.

Pillar Design Dimensions and the Depth of Deposition

KGHM Cuprum indicated that like the LGOM mines there is no need in the Kamoa deposit to change the controlled convergence room and pillar systems with an increase in the depth of deposition.

When using the controlled convergence room and pillar system the destressed and delaminated zone is created above the extraction panel and only the rocks located within this destressed area constitute a load for the primary pillars. The depth of the deposit does not therefore have an essential impact on the geometry of the primary pillars. The primary pillars are designed to ensure this load will destroy their structure and enable the deformation of the entire overlaying strata.

The size of the pillars depends mainly on thickness of the deposit, the strength parameters of the rocks in the deposit profile and the structure of the roof rocks.

When underground exposures at the Kamoa deposit become available the current ground conditions and structural geology assumptions must be verified with structural and geotechnical mapping and geotechnical instrumentation programmes to observe and further understand rock mass behaviour to ensure the successful implementation of the controlled convergence room and pillar system.

Proposed Controlled Convergence Room and Pillar Extraction Ratios for the Kamoa Deposit

KGHM Cuprum calculated the proposed extraction ratios based on analysis of the geological structure of the Kamoa deposit and the surrounding rock mass, data obtained from the drilling cores, the results of numerical modelling and the experiences gained from mining of the LGOM sedimentary deposit that is geological comparable to the Kamoa deposit.

The calculation of the extraction ratios included determining design widths for main and preparatory working protection pillars and the expected final extraction ratios for these respective protection pillars.

The deposit yields using the controlled convergence room and pillar system are presented in Table 16.23.

Table 16.23 Controlled Convergence Room and Pillar- Deposit Yields (Extraction Ratios)

Depth	Deposit thickness	Yield in the exploitation panel	Yield after mining the pillars of preparatory workings
Up to 600 m	4 m	93.4%	91.5%
	6 m	90.6%	88.1%
Up to 600 m (Deposit dipping ≥12.5°)	4 m	90.1%	88.2%
	6 m	85.9%	83.4%
Greater than 600 m	4 m	89.2%	87.3%
	6 m	86.0%	83.5%
Greater than 600 m (Deposit dipping ≥12.5°)	4 m	84.8%	82.9%
	6 m	79.0%	76.5%

Recommendations on Controlled Convergence Room and Pillar

There is a significant difference between the KGHM Cuprum extraction ratios based on the controlled convergence room and pillar model and the previous extraction ratios developed for the Kamoa 2016 PFS.

The significantly improved extraction ratios delivered by the KGHM Cuprum proposed controlled convergence room and pillar method indicates the need to further assess the merits of adopting this mining method for the Kamoa deposit.

16.6 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 2016 PFS 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

17.1 Introduction

This section on recovery methods incorporates assumptions, analysis and findings of the Kamoa 2016 PFS.

The process plant consists of a 3 Mtpa Run of Mine (ROM) concentrator based on staged crushing, ball mill grinding and flotation. The output of the process plant is copper concentrate which is sold. The basis of design for the concentrator is outlined in Table 17.1.

Table 17.1 Process Plant Basis of Design

Production Information	Unit	Detail
Monthly Throughput	t/month	250,000
Annual Throughput	t/a	3,000,000
Operating Days	days	365
Utilisation	%	87
Run Hours	h/a	7,621
Plant Throughput	t/day	9,447
Plant Throughput	t/h	394
Feed Grade Cu	%	4.24
Recovery	% Cu	86
Collector Dosage	g/t milled	156
Promoter Dosage	g/t milled	28
Frother Dosage	g/t milled	95
Flocculant Dosage	g/t milled	35

17.2 Process Description

All underground ore 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 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.

Three variable speed apron feeders are available to recover ore from the stockpile and feed to the primary crusher. Two feeders will typically operate and one will be on standby or under maintenance. 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 third metal detector is positioned immediately ahead of the primary crusher feed bin. A variable speed vibrating feeder at the base of the bin feeds the primary crusher. Primary crushed ore joins secondary crushed ore and is sent to the three sizing screen feed bins. Each bin has a variable speed vibrating feeder to feed the three 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 two secondary crusher feed bins has a vibrating feeder, each feeding a cone crusher. Both secondary cone crushers are expected to be in operation, but not full time.

Sizing screen undersize is sent to the mill feed stockpile. The undersize is nominally -8 mm to minimise the potential for scatting (discharging unground oversize) and maximise grinding efficiency in the primary mill. The mill feed stockpile is covered to minimise dust and has four vibrating feeders below it that feed ore onto the mill feed conveyor. The mill feed conveyor also has a grinding ball feeder discharging onto it to maximise primary milling power.

The primary ball mill is designed to conduct the coarse grind component only and will reduce the ore to about 150 µm P_{80}. Dry mill feed from the conveyor falls into the mill feed chute where water is added and 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 direct 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 primary mill discharge sump.

New feed to the secondary milling circuit together with secondary mill discharge is fed to the secondary cyclone cluster by one of two variable speed pumps. Cyclone underflow feeds the secondary ball mill together with the majority of the flotation collectors. 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 rougher flotation feed conditioning tank. The flotation feed stream is sampled for accounting purposes.

The flotation feed is pumped from the conditioning tank via two pumps (one operating, one standby – both Variable Speed Drive, VSD) to the rougher flotation bank. Frother and collector are added at the feed box to the first rougher flotation cell and can be added to subsequent rougher and scavenger cells. Rougher concentrate from the first two cells is pumped (duty and standby) to rougher cleaning cells. Rougher tails gravitates to the first scavenger cell.

Scavenger flotation takes place in a bank of nine cells. Scavenger concentrate forms part of regrind mill feed and scavenger tails forms part of the final tails stream.

Rougher cleaner concentrate is sent to rougher recleaner flotation and part 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 mill or mills (relatively recent IsaMill testwork indicates that a single regrind mill may suffice while the capital cost estimate is based on two of the same 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.

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 with flocculant in the concentrate thickener feed tank then sent to the thickener feed well by gravity. Thickener overflow reports to the concentrate thickener overflow tank, from where it is redistributed around the flotation circuit as spray water to maximise reagent reuse. Excess thickener overflow reports to the process water tank. 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 and the slurry flows by gravity to the tailings thickener feedwell. 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.

Table 17.2 Design Criteria

	Unit	Value (Design)
Annual Plant Feed	t/y	3,000,000
Overall Crusher Availability	%	65
Crusher Operating Time	h/y	5694
Crushing Circuit Feed Rate	t/h	527
Overall Mill Availability	%	87
Mill Operating Time	h/y	7621
Milling Circuit Feed Rate	t/h	394

These availability figures are in line with industry norms for these types of operations after incorporating allowances for local issues such as power reliability.

17.3 Concentrator Basis of Design

The concentrator design is based on expectations for the first nine years of operation (Table 17.3). The ROM feed is taken to be 87% Hypogene and 13% Supergene based on the total production schedule average. Appropriate design margins have been incorporated.

Table 17.3 Concentrator Basis of Design

Option	Units	Value	Comment
Flotation Feed	Mt/y	3	–
Average Feed Rate	t/h	394	–
Maximum Feed Rate	t/h	433	+10%
Average Feed Grade	%Cu	3.84	Kamoa 2016 PFS Mine Plan
Early Ore Grade	%Cu	4.24	Average Years 1–9
Design Feed Grade	%Cu	5.14	Max Annual Average Grade years 1–9 plus 7.5%
Relative Abundance - Hypogene (%)	Mass %	87	Kamoa 2016 PFS Mine Plan
Relative Abundance - Supergene (%)	Mass %	13	Kamoa 2016 PFS Mine Plan
Concentrate Grade	%Cu	37	Lower than mine production assumption, see text
Copper Recovery	%	86	From testwork
Design Mass Pull	Mass %	11.93	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 is 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 also a legitimate 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 Kamoa Crushing and Milling

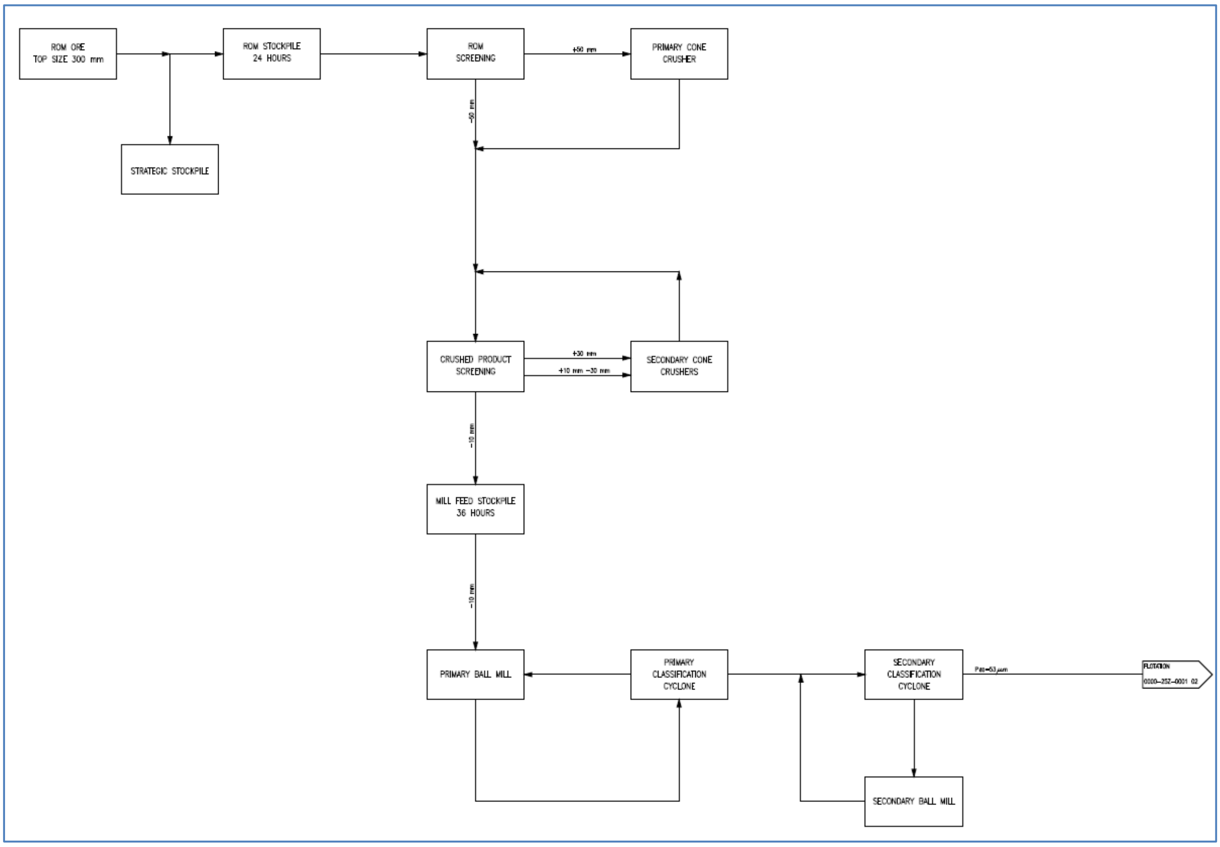


Figure Courtesy MDM 2015

The block flow diagram for flotation, concentrate handling and tailings is shown in Figure 17.2.

Figure 17.2 Kamoa Flotation Circuit

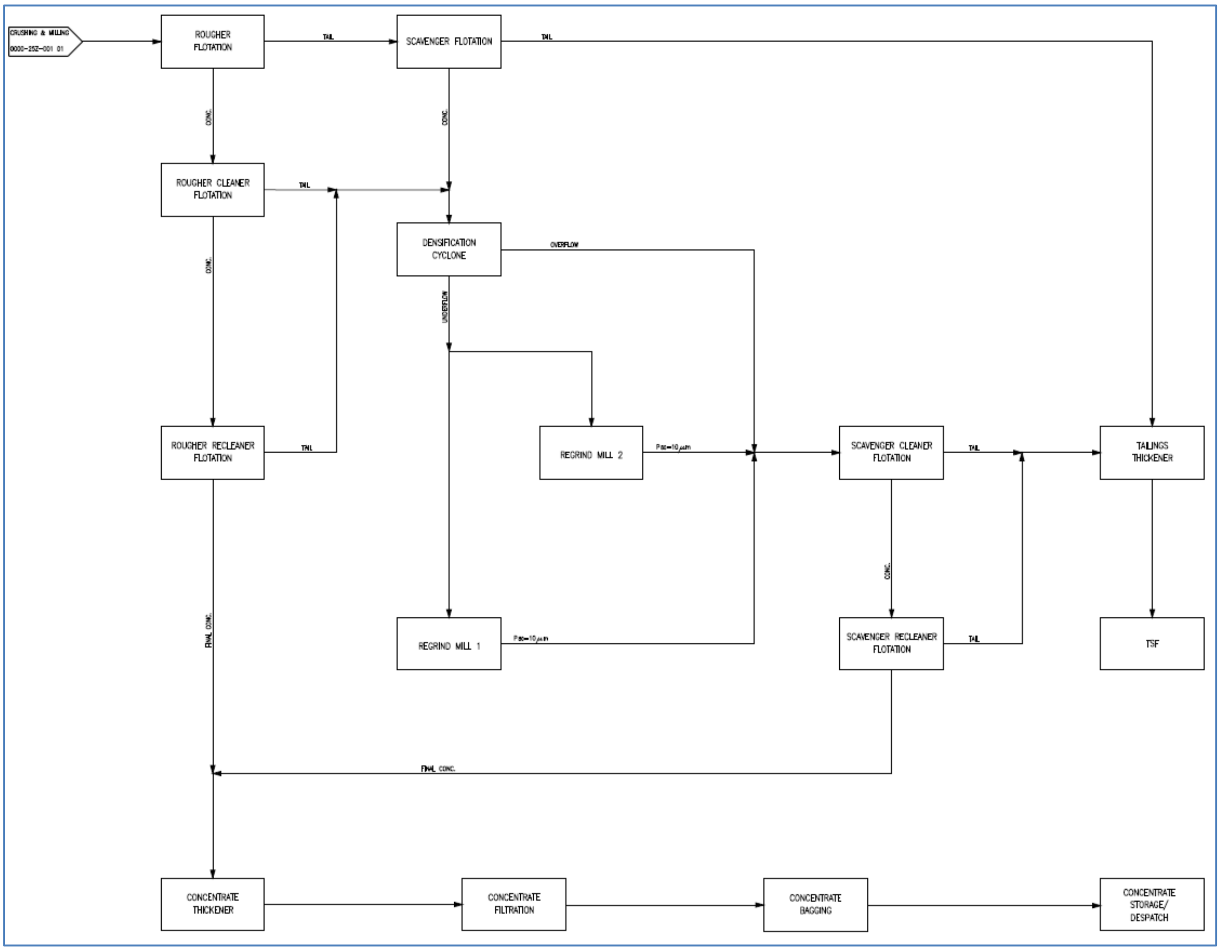


Figure Courtesy MDM 2015

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 toa 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 filter presses. 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 17.4 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 17.4 Concentrator Equipment Requirements Summary

Item	Description	Size/Capacity	No. Required + standby	Power Installed kW per unit
Crushers	Primary cone	CS660	1	315
	Secondary cone	CH865	2	500
Screens	Primary	2.4 m x 4.27 m	1	45
	Secondary	3.1 m x 6.1 m	3	55
Mills	Primary Ball Mill	22 ft x 36 ft	1	7000
	Secondary Ball Mill	22 ft x 36 ft	1	7000
	Concentrate regrind	IsaMill M10000	2	3000
Cyclones	Primary cluster	750 mm Diameter	3 + 1	500 (1+1 feed pump)
	Secondary cluster	420 mm Diameter	7 + 1	355 (1+1 feed pump)
	Concentrate regrind cluster	100 mm Diameter	18 + 4	75 (1+1 feed pump)
Blowers	Flotation air	33 900 Nm^3/h @ 150 kPa	2 + 1	110
Flotation cells (includes agitators)	Rougher	200 m^3	2	250
	Scavenger	200 m^3	9	250
	Rougher cleaner	30 m^3	4	75
	Rougher recleaner	20 m^3	5	55
	Scavenger cleaner	100 m^3	9	225
	Scavenger recleaner	20 m^3	4	55
Thickeners	Concentrate	15 m Diameter	1	11
	Tailings	30 m Diameter	1	18
Filters	Concentrate	Larox PF 132/144	2	18.5
Tailings Pumps	Centrifugal	426 m^3/h	4 + 4	250

Table 17.5 lists the estimated projected water, consumables and power requirements for the concentrator.

Table 17.5 Projected Concentrator Water, Power, and Consumables

Item	Description	Units	Consumption units/t Plant Feed	Annual Requirement
Power	Electric	kWh	66.9	201 GWh
Water	Raw make-up	m^3	0.43	1280 Mm^3
Reagents	Frother	g/t	95	285 t
	Collector	g/t	156	468 t
	Promotor	g/t	28	84 t
	Flocculant (Tailings and Concentrate)	g/t	35	105 t
Consumables	Grinding media (75 mm steel balls)	kg/t	0.98	2940 t
	Grinding media (35 mm steel balls)	kg/t	1.98	5940 t
	Grinding media (2 mm Ceramic)	g/t	137	411 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 17.6.

Table 17.6 Processing Production Schedule

Description	Unit	Total	Project Time (Years)											
			1	2	3	4	5	6	7	8	9	10	11	12
Ore Milled	(kt)	71,893	2,670	3,000	3,000	3,009	2,990	2,993	3,001	3,008	3,007	3,030	3,032	3,028
Cu Grade Milled	(%Cu)	3.86	3.87	3.92	4.39	4.78	4.74	4.16	4.25	4.19	3.90	3.88	3.85	3.67
Concentrate Produced	(kt)	6,106	228	259	293	321	316	276	283	279	259	259	257	244
Concentrate Cu Grade	(%Cu)	39	39	39	39	39	39	39	39	39	39	39	39	39
Cu in Concentrate	(kt)	2,394	89	102	115	126	124	108	111	109	101	102	101	96
Cu in Concentrate	(Mlb)	5,277	197	224	253	278	273	238	244	241	223	224	222	211

Description	Unit	Total	Project Time (Years)											
			13	14	15	16	17	18	19	20	21	22	23	24
Ore Milled	(kt)	71,893	3,012	3,004	3,014	3,015	3,011	3,002	2,998	3,003	3,019	3,002	3,030	3,015
Cu Grade Milled	(%Cu)	3.86	3.60	3.95	3.94	3.45	3.50	3.50	3.73	3.79	3.43	3.29	3.70	3.04
Concentrate Produced	(kt)	6,106	238	261	262	227	231	230	246	251	226	215	246	198
Concentrate Cu Grade	(%Cu)	39	39	39	39	39	39	39	39	39	39	39	39	39
Cu in Concentrate	(kt)	2,394	93	102	103	89	90	90	96	98	89	84	97	78
Cu in Concentrate	(Mlb)	5,277	206	226	226	197	199	199	212	217	196	186	213	171

The schedule allows for a rapid ramp-up of the concentrator in year 1 and consistent treatment rates after that.

17.6 Comments on Section 17

ROM ore is assumed to have a topsize of 300 mm controlled by intensive blasting and 250 mm square grizzly installations at each ore dump point underground. If this topsize 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 in excess of 300 mm are will cause problems for the conveying system that brings the ore 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 reasonably robust but there remains substantial variability in gangue recovery to concentrate. Further investigation of silica rejection is required to ensure consistent concentrate quality from the plant. In addition, the fine rougher grind size (P_{80} 53 µm) has implications for mine backfill that need further investigation in the next project stage, should backfill still be necessary.

The flotation circuit configuration deliberately avoids recycle streams in accordance with the XPS testing philosophy. This results in theoretically well-defined residence times throughout the circuit but does present a risk with regard to managing varying ore grade and copper sulphide mineralogy. The most likely streams to be recycled in the current configuration are the rougher recleaner tail (to rougher cleaner feed) and the scavenger recleaner tail (to scavenger cleaner feed). Flowsheet provisions for these 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. The mineralisation is highly variable and further work is needed to better define mineralogy in the various parts of the Kamoa deposit. High quality mineralogical information will be necessary for feed grade blending and controlling final concentrate grade.

In the dominant hypogene ores there is some uncertainty is with respect to concentrate grade, which will vary with chalcopyrite to bornite proportion and the amount of silica recovered to concentrate. Hypogene ore recovery is always high (95% in roughers and 90% overall) as there is little unfloatable copper mineralisation present. Improved hypogene ore definition requires identification of the relative proportions of chalcopyrite, bornite and chalcocite.

Supergene has broad variability in both the relative proportions of the floating copper minerals and the proportion of copper that will be lost to tailings in non-floating minerals. This leads to uncertainty with regard to both recovery and grade of copper concentrate. . Improved supergene ore definition requires identification of the relative proportions of chalcopyrite, bornite, chalcocite 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 impact of the supergene uncertainty is important, but it is not felt to be material to the project economics as supergene only represents 13% of the concentrator feed in the PFS mine plan.

18 PROJECT INFRASTRUCTURE

18.1 Introduction

This section describes the project infrastructure work that has been carried out to date and the scenario that has been developed for the Kamoa 2016 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 concentrate will be transported via road to Ndola in Zambia and thereafter via rail to Durban harbour in South Africa.

Power for the Kamoa 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 Kamoa will be powered by electricity sourced from the grid and on-site diesel generators.

18.2 Site Plan and Layout

A site plan (Figure 18.1) displays the locations of the proposed plant site, and closely associated facilities.

Figure 18.1 Site Conceptual Infrastructure Layout Plan

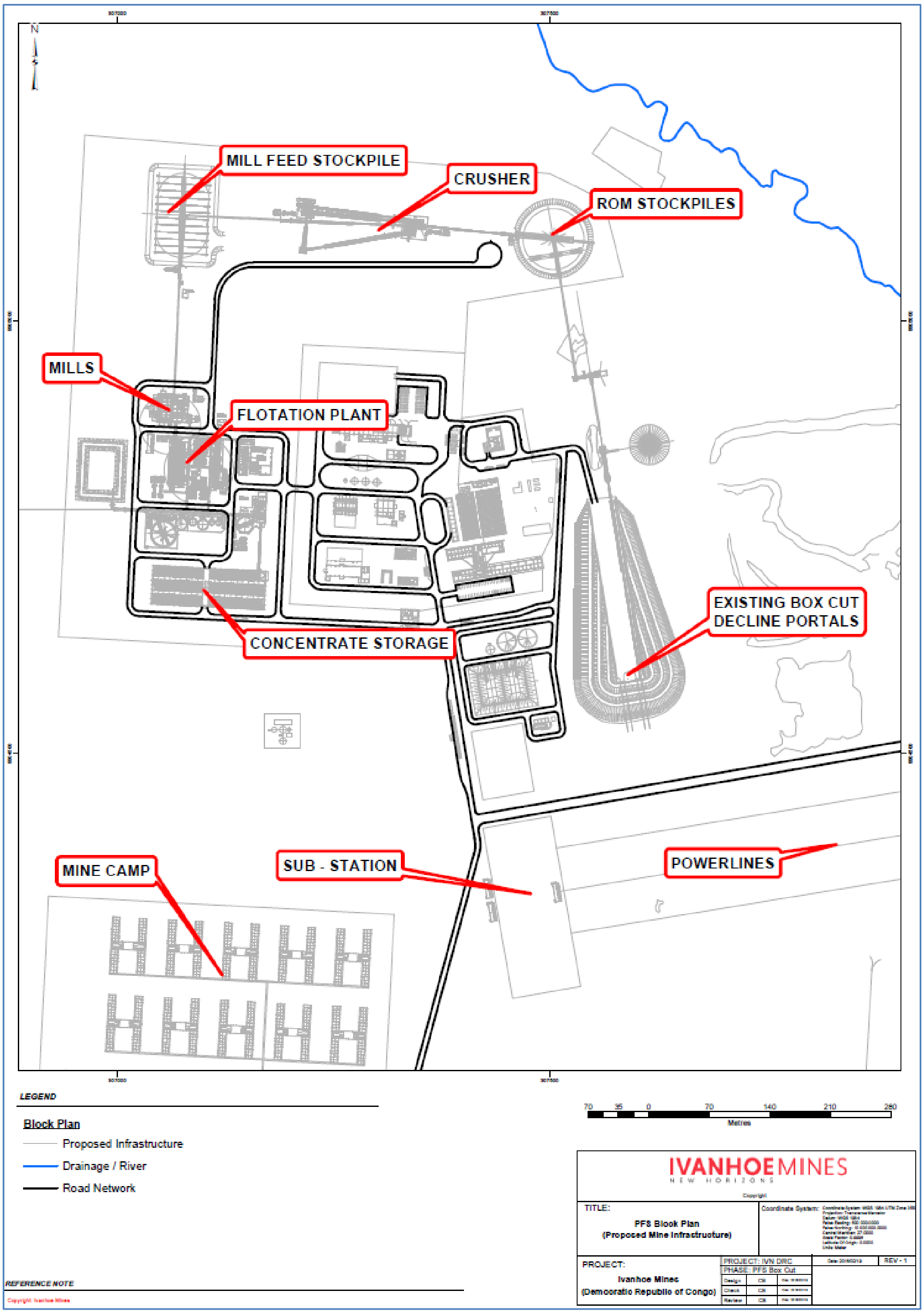


Figure by Ivanhoe 2015.

The site is compact and incorporates all comminution and concentration facilities, reagent preparation, laboratory, offices, construction camp, electrical infrastructure, water infrastructure, surface mining offices and workshops, vehicle parking, warehouse storage and lay-down facilities.

Of note in the above diagram is the proportionally large area for concentrate storage. Unlike large low grade copper operations, a major proportion of the mined ore (about 10%) is recovered to concentrate given head grades in the region of 4% Cu.

All infrastructure has been incorporated in the capital cost estimate.

18.3 Power

18.3.1 Generation

Power for the Kamoa project is planned to be sourced from the Democratic Republic of Congo's state-owned power company (SNEL, Société Nationale d'Electricité) electrical interconnected grid. This electrical grid faces a huge deficit of power production due also to old hydropower plants with broken turbines to be completely renewed.

The three out of four hydro power plants in the SNEL southern grid that are considered in the Ivanhoe Mines-SNEL power project are: Koni, Mwadingusha and Nzilo 1. All three require refurbishing. The three plants combined could produce over 200 MW. Prior to completion of the refurbishments, 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 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). After rehabilitation of these two hydro power plants for the first phase mine development, SNEL guarantees to Kamoa 100 MW. As well as the generating plant refurbishment, a new 220 kV substation and the alignment for the new 220 kV high-voltage line to the Kamoa 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 line (20 km) will be built from a point from the R0-Kisenge line at 20 km from the Kamoa mine.

A 120/11 kV, 15 MVA mobile substation has also been provided in the process plant capital estimate to feed construction power. The line and substation will be retained as emergency back-up power supply.

Prior to completion of the overhead power line, a diesel generated power plant will be implemented in order to provide power during the construction and start-up phases of the Kamoa Project. This power plant will subsequently be maintained as emergency back-up power supply.

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 108 megawatts (MW).

Figure 18.2 shows the locations of the existing power plants in relation to the Kamoa site. Figure 18.3 shows the state of electrical powerline infrastructure in proximity to the Project area.

Figure 18.2 Power Plants Locations

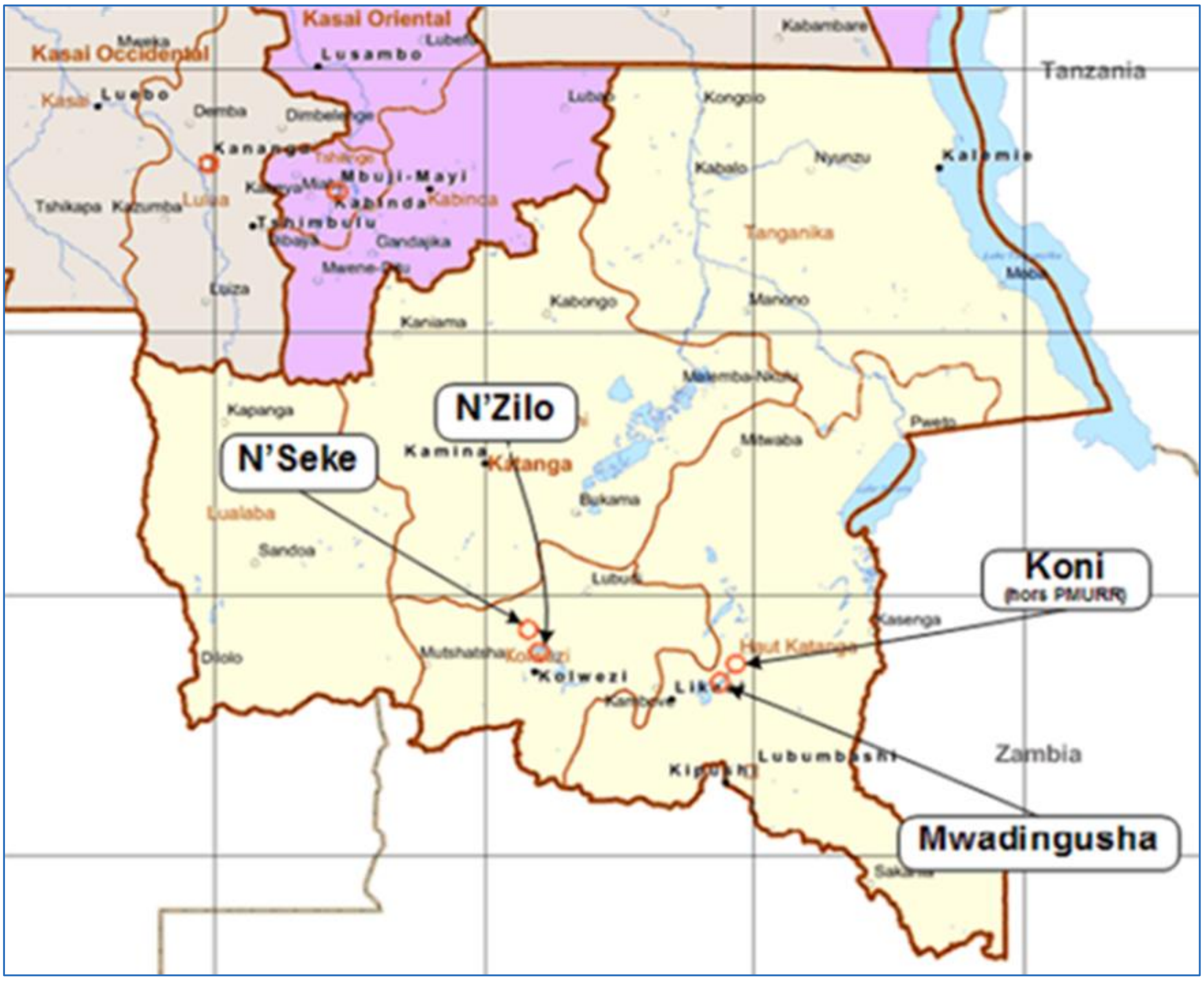


Figure by AMEC, 2012. Map north is to top of plan. Grid squares on plan indicate scale and are approximately 250 km x 250 km.

Figure 18.3 Example Existing Power Transmission Lines in Proximity to Kamoa Site

Photograph by Ivanhoe, 2012.

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, only turbines five and six, are currently operational.

Koni Hydroelectric Power Plant

Koni is located 7 km upstream 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 Mwadingusha hydro power plant is located on the Lualaba River, approximately 30 km from the city of Kolwezi in the province of Lualaba in the DRC. 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. All of these 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 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. The OPGW (optical ground wire) will be installed to the two Nzilo-RO 120 kV lines (20 km).

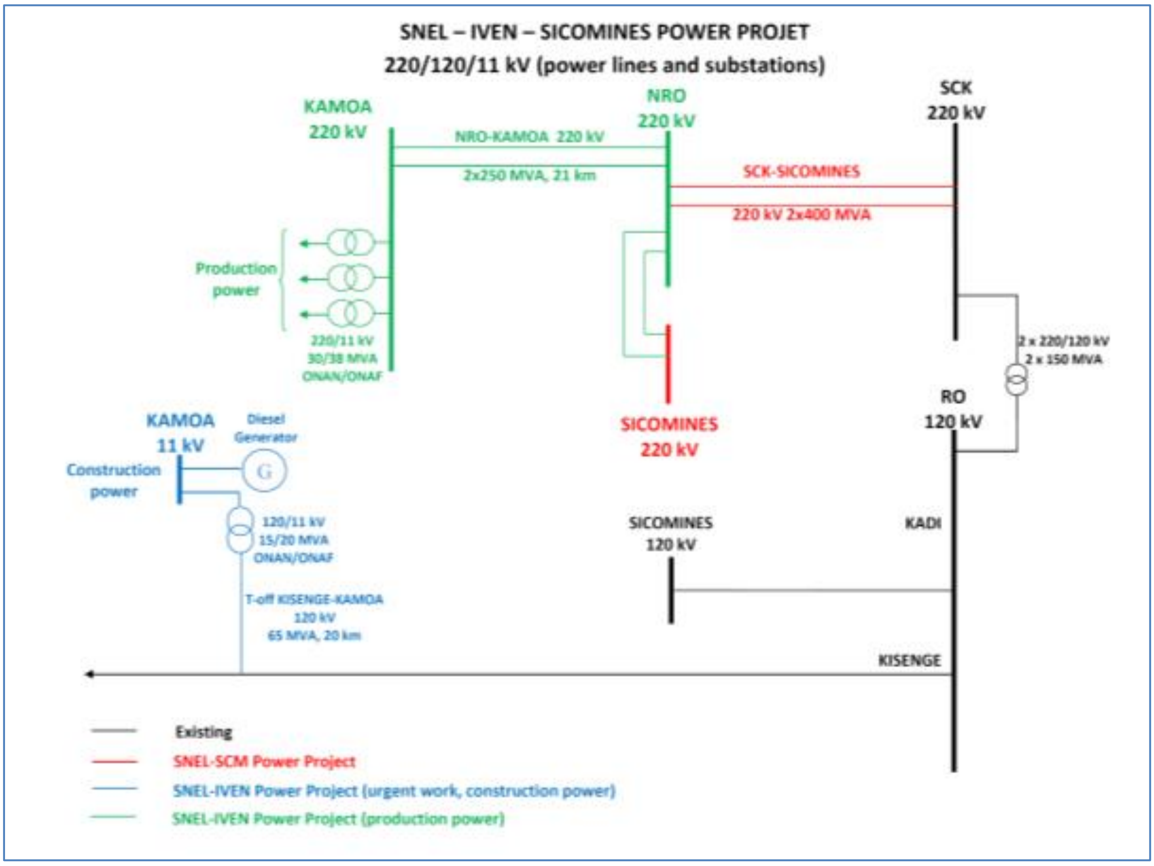
The two 120 kV bays at RC substation where Koni and Mwadingusha power feeds to the SNEL grid will be also refurbished.

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 Kamoa mine for construction power, through a 120/11 kV, 15 MVA mobile substation to be installed at Kamoa 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 Kamoa mine substation. A new 220 kV repartition substation (named NRO) will be built at the SNEL side.

The Figure 18.4 shows the design of new transmission lines and substations.

Figure 18.4 Planned Transmission Lines and Substations



18.4 TSF

As part of the Kamoa 2015 PFS, Epoch Resources (Pty) Ltd (Epoch) was appointed by Ivanhoe to undertake the PFS Design (PFD) of the Tailings Storage Facility (TSF) and its associated infrastructure.

The terms of reference that Epoch was responsible for included:

- The design of the Tailings Storage Facility (TSF) comprising a 24 year capacity, return water sump and any associated infrastructure
- Estimation of the capital costs, operating costs and closure costs
- Estimation of the costs over the life of the facility.

The site selection study undertaken in 2014, by Epoch, found the most favourable site as being the Mupenda site.

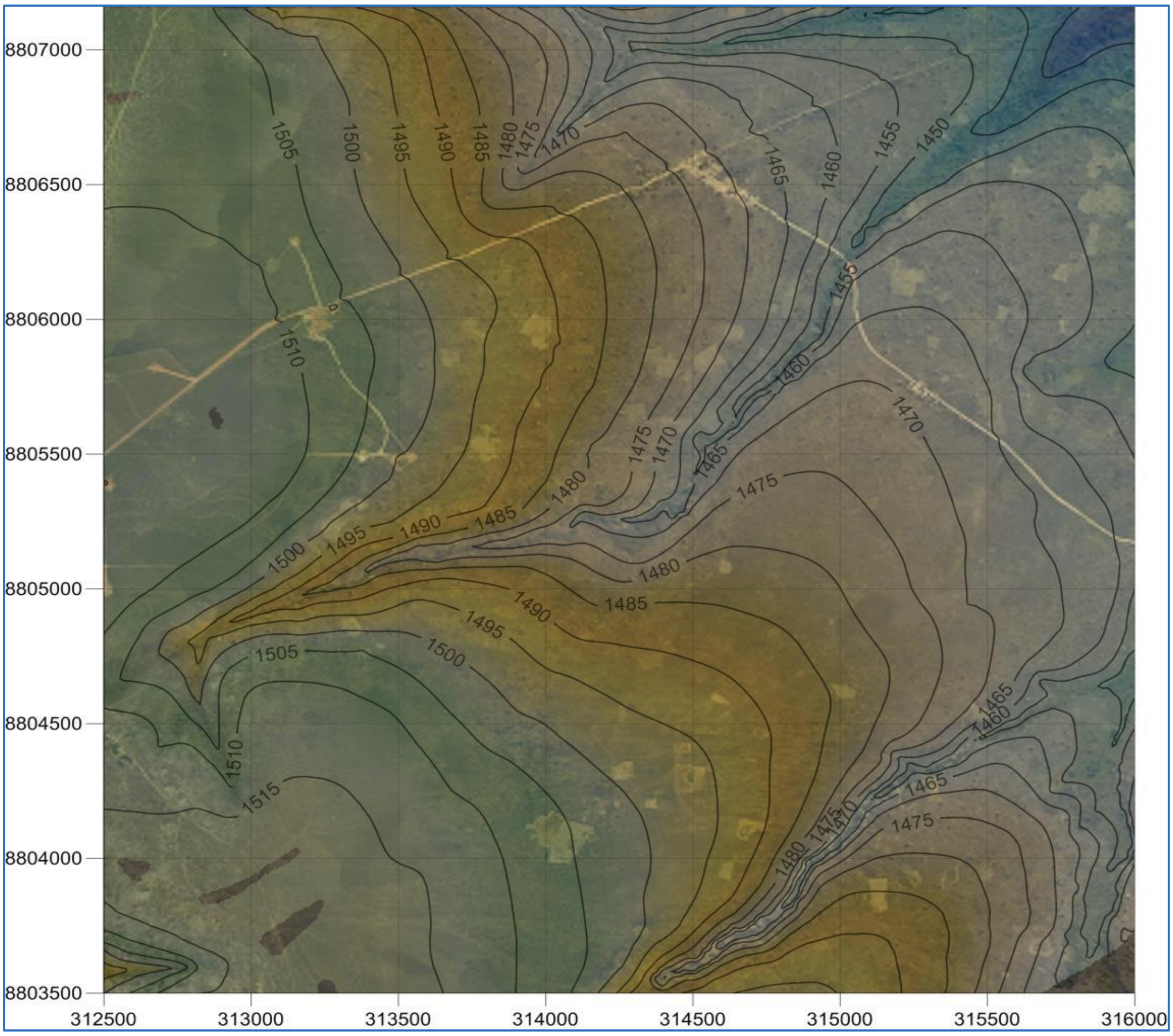
The key design features of the TSF are as follows:

- It will be constructed in 5 phases as a 257 ha valley impoundment dam with a downstream compacted earth embankment wall having a maximum height of 42 m.
- A concrete lined Return Water Sump with a water storage capacity of 1,000 m^3; and
- A spigotted slurry delivery 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 18.5.

Figure 18.5 General Topography of the Preferred Kamoa TSF Area



18.4.1.1 Terms of Reference

Epoch are responsible for:

- The design of the TSF comprising:
  - A TSF (TSF) that accommodates 45,678,000 dry tonnes of tailings over the 24 year life;
  - A Return Water Sump (RWS) associated with the TSF; and
  - The associated infrastructure for the TSF (i.e. perimeter slurry deposition pipeline, storm

water diversion trenches, perimeter access road etc.);

- Estimation of the capital costs to an accuracy of +/-25%, operating costs associated with these facilities to an accuracy of +/-25% and closure costs to an accuracy of +/-35%; and;
- Estimation of the costs over the life of the facility.

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 Kamoa's TSF

Description	Value	Unit
Design Life of Facility	24	years
Processed Ore	Copper	
Tailings Deposition Rate:		
Year 1 – Year 3	2,800,000	dry tpa
Year 4 – Year 24	1,700,000–1,800,000	dry tpa
Particle Specific Gravity	2.85	
In-Situ Void Ratio	1	
Particle Size Distribution	100% passing 20 µm sieve	
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 Boxcut construction for the mine decline and within the TSF basin;
- The legislation that has been adopted for the purpose of this study is "Appropriate Best Practise Measures". "Appropriate Best Practise Measures", in this case, implies the use of the South African TSF Design Standards and Codes (i.e. SANS 0286:1998 – "Code of Practise for Mine Residue") 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; and

- 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 $1x10^{-6}$ cm/s must be lined with an appropriate liner system, according to the DRC Regulations.

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 actual rainfall information was available for the Kamoa site. The Kolwezi rainfall and evaporation data was sourced from the Kolwezi weather station.

The data available comprised:

- Average monthly lake evaporation values; and
- Storm rainfall depths, of 1–7 day duration, for recurrence intervals of 2, 5, 10, 20, 50, 100, and 200 years.

The average monthly rainfall and evaporation values for Kolwezi are summarised in Table 18.2 and the design storm depths are listed in Table 18.3.

Table 18.2 Average Monthly Rainfall and Lake Evaporation Values for Kolwezi

Month	Average Rainfall (mm)	Lake Evaporation (mm)
January	196	120
February	171	112
March	225	77
April	79	75
May	14	100
June	1	69
July	0	61
August	4	50
September	23	63
October	91	88
November	193	98
December	209	121
Annual	1207	1034

Table 18.3 Design Storm Rainfall Depths for Kolwezi

Duration (Days)	2 Years	Rainfall Depth (mm) For Each Recurrence Interval					200 Years
		5 Years	10 Years	20 Years	50 Years	100 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 1000 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 grading's 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 soil layer along the valley which does not provide the necessary permeability requirement to adhere to the DRC Regulations. This soil, termed Kalahari Sands, was found to have a permeability higher than the limit of 1×10^{-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 have 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 undertaken 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 based on the following criteria:

- Suitable topography for storing the require capacity;
- The nature of the material to be deposited and its short and long term environmental impacts;
- The nature and sensitivity of the surrounding environment, i.e. the receiving environment;
- The underlying soil properties, i.e. the need for liners/compaction;
- A risk analysis was performed, which took cognisance of the environment, people and nearby infrastructure; and
- High level costs associated with the construction and operation of the TSF.

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;
- 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 a failed TSF, as well as little to no people residing downstream of the site; and
- High level costs showed that this site would have the lowest costs to construct a TSF.

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; and
- 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 Rate of Rise (RoR) of 0.5 m/year. This was due to the fineness of the tailings (80% passing 50 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 (100% passing 20 µm). 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:

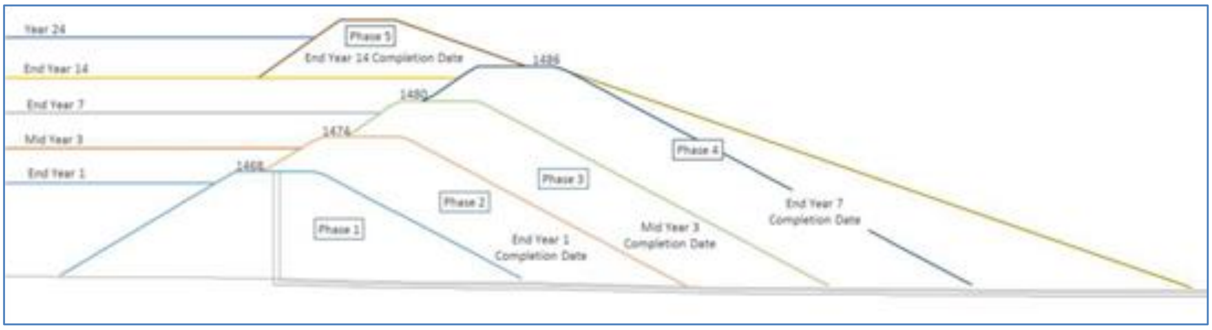
- Subsoil drains cannot be constructed in Kamoa's tailings as they will blind and become inoperable. A full containment facility can utilise a curtain drain to reduce the phreatic surface through the wall. A curtain drain will not be in contact with the tailings and thus cannot blind;
- Slurry water 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; and
- 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.

18.5 TSF Alternative Design

The design of the TSF has called for a downstream facility, thus each successive lift of the wall requires ever increasing volume of borrow material. It may be possible to convert the TSF to an upstream facility once it has reached an acceptable rate of rise and the tailings adjacent to the wall has achieved sufficient strength. Borrow material will still be used to construct the final Lift of the wall, however in an upstream direction (see Figure 18.6). This will greatly reduce the volume of borrow material (approximately 3,000,000 m^3) and therefore the costs.

Figure 18.6 Converting to Upstream Facility in Phase 5



In order to determine whether this is feasible, stability modelling of this option must be underaken, as well as field investigations during operation of the facility. Therefore, the information required to make a decision to move to an upstream facility may only be available during operation of the facility.

18.6 TSF Construction

The TSF has been designed to accommodate a volumetric storage capacity of 45,678,000 dry tonnes, corresponding to 32,627,000 m^3, over a 24 years and comprises the following facilities:

- A TSF;
- A Return Water Sump;
- Associated infrastructure (i.e. slurry delivery infrastructure, storm water diversion trenches, etc.).

The key design features of the TSF are:

- The TSF will be constructed as a valley impoundment dam with a compacted earth embankment wall. This will have the following features:
  - The TSF will be constructed as a downstream facility;
  - The wall is to be raised in 5 phases, where Phase 1 is at elevation 1468 MAMSL and the last phase is at elevation 1492 MAMSL; and
  - The TSF has a total footprint area of 257 ha, a maximum height of 42 m and a final rate of rise of <1 m/year;

- A concrete lined RWS with a water storage capacity of 1,000 m³; and
- A slurry spigot pipeline along the crest of the TSF.

18.6.1.1 Stage Capacity and Site Development Strategy

The facility will be constructed as a downstream TSF and as such requires careful planning of each lift of the impoundment wall.

An initial 18 m compacted earth starter embankment corresponding to a crest wall elevation of 1468 MAMSL, will be constructed to yield one year's capacity for tailings. Thereafter the construction of the wall phases must be at least 2 m ahead of the tailings to allow for sufficient freeboard.

The 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), three intermediate walls will be provided at the upstream side of the valley. This will hold the tailings back while allowing the contractor to construct the penstock and liner. The penstock, liner and intermediate walls will have three stages which correspond to certain impoundment wall phases, as discussed later.

The staged development of the TSF after year 1 and year 24 of operation is shown in Figure 18.7 and Figure 18.8 respectively.

Figure 18.7 Development of the Kamoa Copper TSF after 1 Year

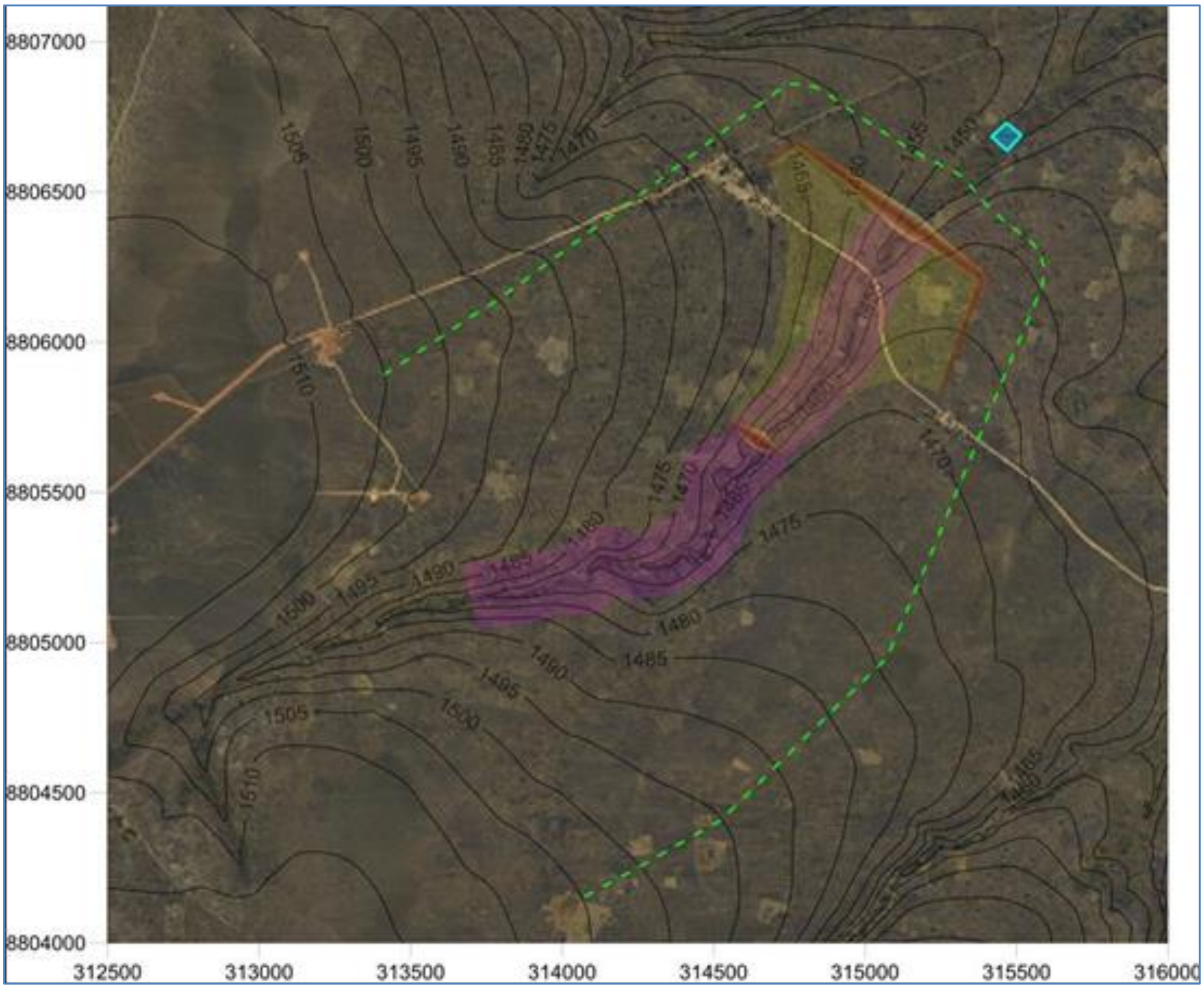


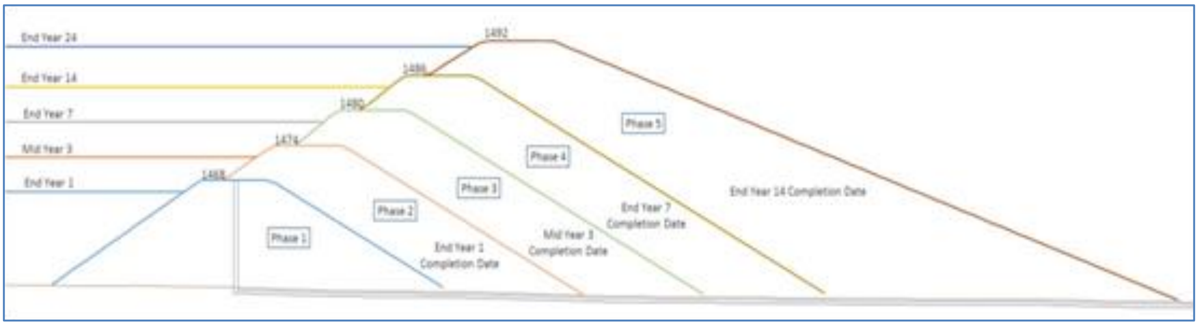
Figure 18.8 Development of the Kamoa Copper TSF at Final Elevation

Table 18.4 shows the tailings elevation criteria used to determine the wall elevations for each phase. Each corresponding impoundment wall elevation and required completion date is also shown. This was determined by allowing 2 m of freeboard at all times. The Table also shows in which phase the intermediate walls will be required. Figure 18.9 shows the impoundment wall phasing graphically.

Table 18.4 Tailings Stage Capacity and Phasing Information

Phase	Tailings Elevation at the end of Phase (MAMSL)	Required Wall Elevation Prior to end of Phase (MAMSL)	Required Completion date of Wall Phase	Intermediate Wall
Phase 1	1466	1468	Start-up	Stage 1
Phase 2	1472	1474	End Year 1	Stage 2
Phase 3	1478	1480	Mid-Year 3	–
Phase 4	1484	1486	End Year 7	Stage 3
Phase 5	1490	1492	End Year 14	–

Figure 18.9 Impoundment Wall Phasing



18.6.1.2 TSF Preparatory Works

The preparatory works associated with the TSF comprise the following:

- Topsoil stripping to a depth of 300 mm beneath the TSF footprint;
- A boxcut to a depth of 500 mm beneath the starter wall embankment;
- A compacted key below the Phase 1 wall embankment comprises the following:
  - Depth required shall be deep enough to remove the Kalahari sands layer;
  - 10.0 m wide;
  - 1V:1.5H side slopes; and
  - 3.5 m wide compacted bentonite-enriched earth layer to prevent excessive seepage under the wall.
- A compacted earth starter wall embankment with the following dimensions:
  - 18 m high (i.e. crest elevation of 1468 MAMSL);
  - 15.0 m crest width;
  - 1V:1.5H internal side slope; and
  - 1V:2H external side slope.

- A Curtain Drain inside the impoundment wall, to reduce the phreatic surface through the wall. This will comprise the following:
  - Starting 1 m below the top of the wall, ending at the base of the wall and 1 m wide. 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 RWS; and
  - Manholes at each outlet pipe to monitor the drain flows.
- A storm water 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; and
  - 1V:1.5H side slopes.
- A storm water diversion channel with its associated cut-to-fill berm wall with the following dimensions:
  - 1.0 m deep;
  - 1.0 m wide; and
  - 1V:1.5H side slopes.
- A buried 900 ND Class 150D spigot-socket precast concrete penstock pipeline 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 the 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 impoundment wall; 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.

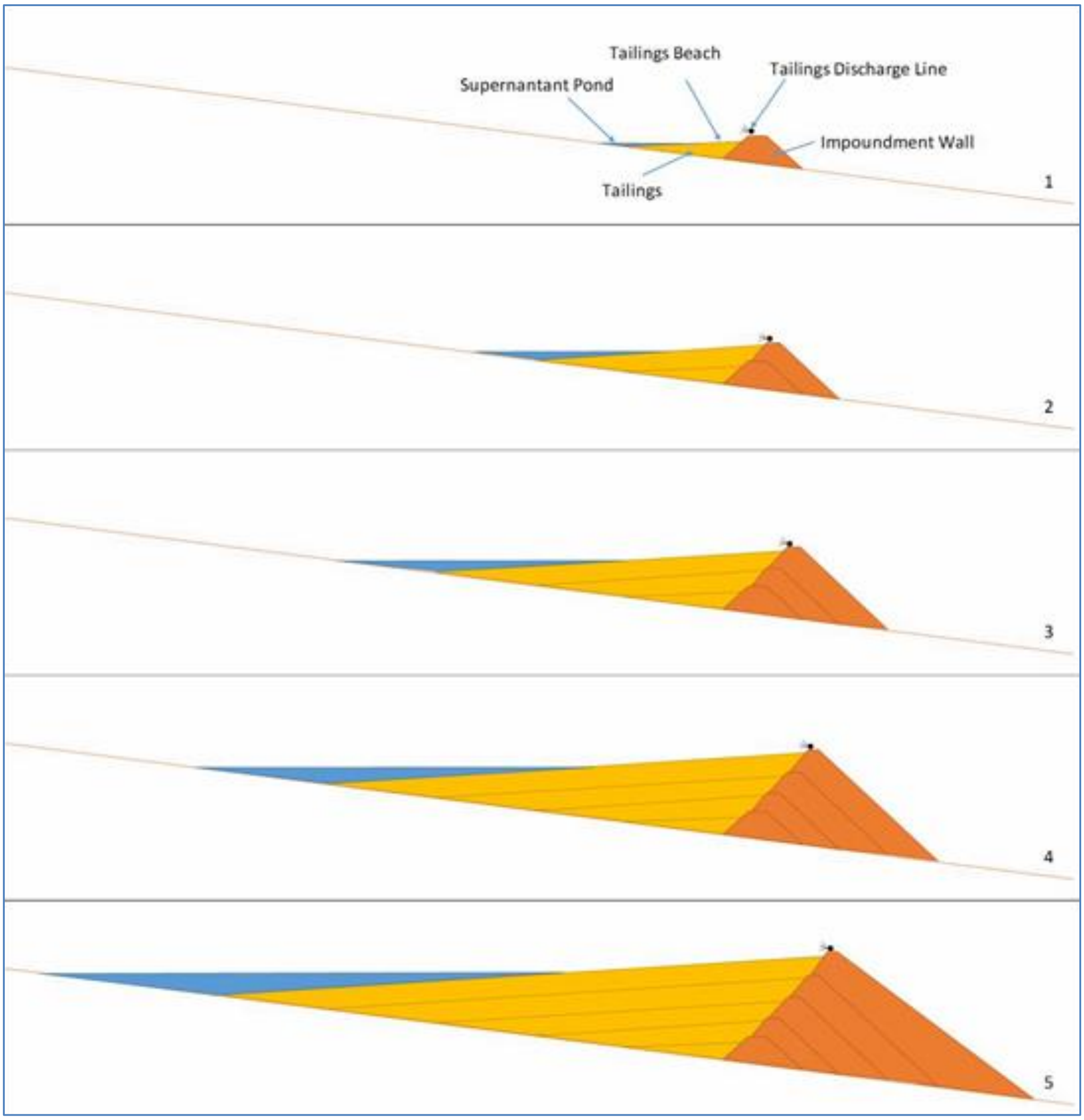
As the impoundment walls will be constantly raised the side slopes have been set to 1V:2H external and 1V:1.5H internal. This both decreases the volume of the wall as well as the time required to construct each lift. The final Phase of the wall will have an external slope of 1V:3H. As no stability analyses 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.

18.6.1.3 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 18.10 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 18.10 Downstream Method of Embankment Construction

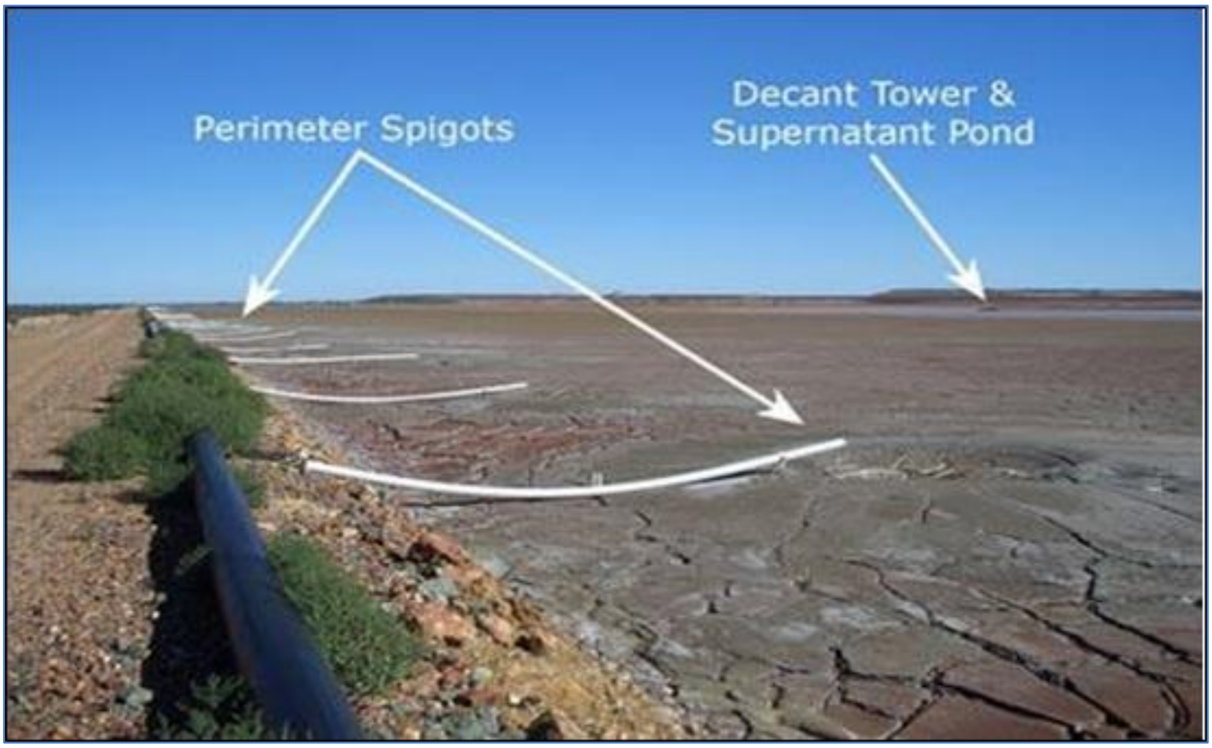


Based on the physical characteristics and actual particle size distribution of the tailings, the following depositional characteristics are affected:

- Tailings behaviour on deposition;
- Beach formation and profile;
- Rate of drying out and desiccation/consolidation and associated strength gain;
- Particle segregation along the beach;
- Pool control; and
- General TSF operation and deposition practices.

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 18.11. 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 18.11 Multiple Spigot Discharge



18.6.1.4 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, and
- The HDPE liner.

These contribute the most to the cost of the project, thus they have been phased.

In order to stage the construction of the penstock and the liner, three intermediate walls will be provided along the valley. This will allow the contractor to construct the penstock and liner while the tailings fills up downstream of the intermediate walls. Figure 18.12 and Figure 18.13 shows where the staging will occur as well as the phase of the impoundment wall each section falls in.

The final Intermediate wall does not affect the staging, however it reduces the length of the penstock pipeline and reduces the area requiring a liner.

Figure 18.12 Cross-Section through Penstock Pipeline Indicating the Penstock and Liner Staging

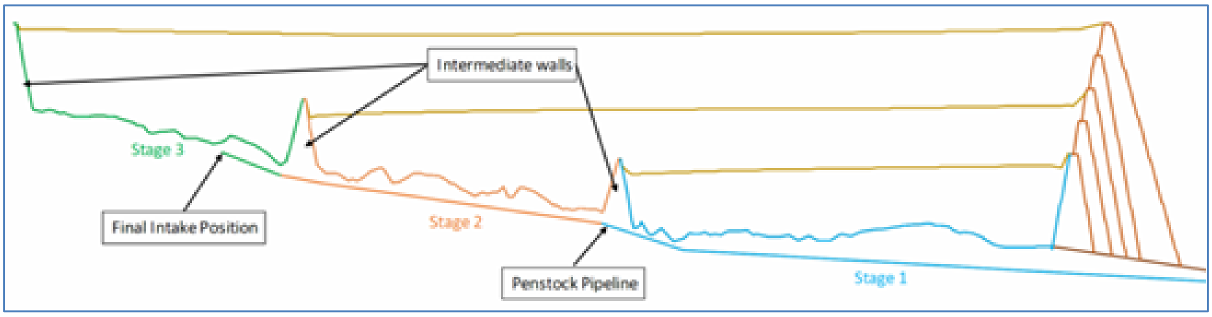
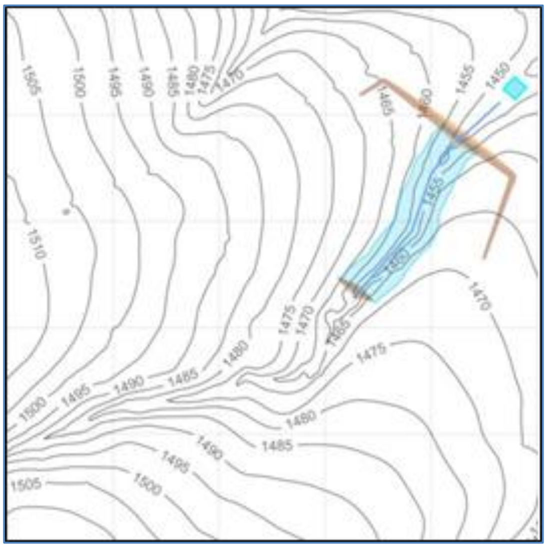
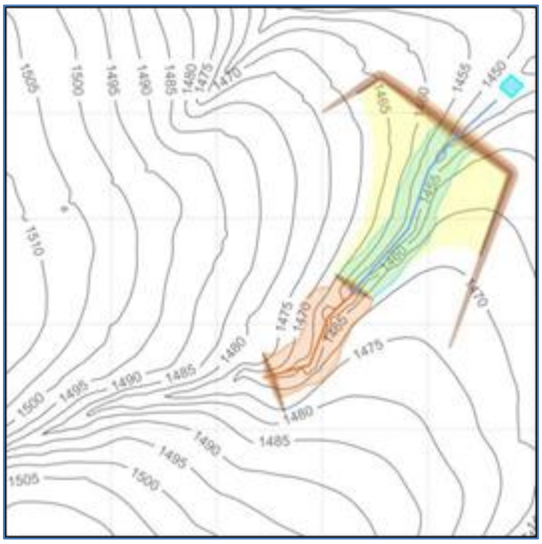


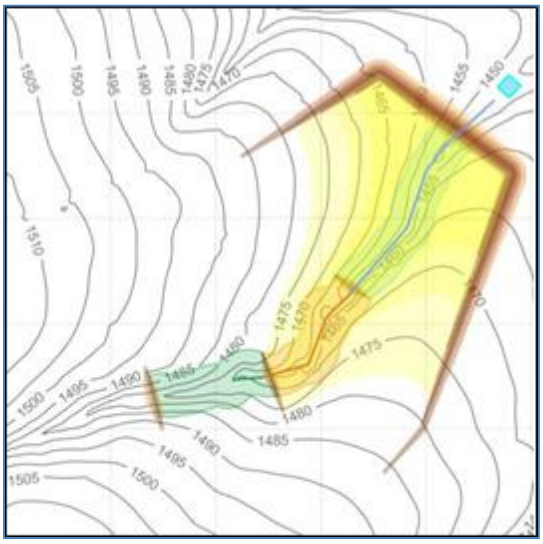
Figure 18.13 Overview of Liner and Penstock Staging



Phase 1 – Start-up



Phase 2



Phase 4

Table 18.5 shows information on the impoundment wall staging. This was determined such that the tailings always has sufficient freeboard.

Table 18.5 Impoundment Wall, Intermediate Wall, Penstock and Liner Phasing

Wall Phase	Penstock, Intermediate walls and Liner	Elevation (MAMSL)	Wall Volume (m^3)	Wall Completion Date	Tailings Elevation at Wall Completion (MAMSL)
Phase 1 – Start-up	Stage 1	1468	290,000	Start-up	1460
Phase 2	Stage 2	1474	500,000	End Year 1	1466
Phase 3	–	1480	880,000	Mid-Year 3	1472
Phase 4	Stage 3	1486	1,400,000	End Year 7	1478
Phase 5 – Final	–	1492	3,300,000	End Year 14	1484

18.6.1.5 TSF Decant System

Supernatant water on the TSF accumulates in a pool as a result of beaching and deposition control. This supernatant water, predominantly derived from the process, but also from rainfall, will be decanted from the surface of the TSF for the following reasons:

- To prevent accumulation and eventually overtopping;
- To reduce infiltration and consequent groundwater contamination; and
- To reduce the potential development of a high phreatic surface with consequent stability problems.

Decant System Trade-off Study

A trade-off study was undertaken to determine whether a floating pumping barge or a conventional gravity penstock system would be selected for the Kamoa. A number of factors were considered in the trade-off in order ensure the correct option was adopted. The following factors were investigated:

- Power Consumption,
- Power Failure,
- Pool Control,
- Settling of Particles,
- TSF Height,
- Emergency measures,
- Ongoing raising of access-way,
- Failure Repairs,
- Expandability, and
- Cost

The penstock system was selected, following the trade-off, for the following reasons:

- Due to the fineness of the tailings stream it was estimated that the beach would have a slope of between 1 in 500 m to 800 m. In other words for a 1 m deep pool, the pool will have a radius of 500 m to 800 m. This size pool can be difficult to control, especially during storm events. Since a barge requires at least 1 m of freeboard, the concern would be that the water could easily accumulate near the wall of the TSF, whereas a penstock can operate at a much shallower water level;
- Since a large volume of water will always be required on the TSF when running a barge, the capacity of the TSF will be limited during storm events, therefore, a Storm Water Dam would be required. A penstock system can operate with a minimal pool size therefore allowing additional capacity during storm events, negating the need for a Storm Water Dam;
- During emergency situations, such as a plant shutdown or a large storm event the penstock system is able to decant large volumes without the restriction of a pumping capacity present in a barge system.

Following this trade-off it was determined that further testing of the tailings will be required to ensure that the correct decant system is selected based on definitive lab results (i.e. Flume tests and rheology tests).

Penstock System

Supernatant and storm water is decanted via vertical penstock intakes and a buried penstock pipeline to the Energy Dissipater and RWS. During commissioning and the initial stages of the TSF development, decanting occurs via temporary/intermediate single penstock inlets located along the migrating path of the pool from the starter wall uphill towards its final location at the double penstock inlets. The intermediate inlets are sealed once the pool has migrated to an adjacent upstream inlet.

The penstock outfall pipe comprises a spigot and socket concrete pipe, encased in concrete. This pipe routes the supernatant water by gravity to a concrete lined energy dissipater structure from where it flows into a Return Water Sump.

18.6.2 Return Water Sump

The RWS has been designed to temporarily store water for pumping back to the plant. As such additional water will be stored on the TSF. Since the TSF is a full containment facility, storing this water should not pose any issues.

The RWS will be concrete lined with twin compartments. The purpose of the twin compartments is to allow desilting of the one while still operating the other compartment. The twin compartments of are each equipped with separate inlet spillways and outlet pipes, which facilitate alternate cleaning and operation of each compartment. A third compartment will be provided for the pumps, with pipes leading out of the twin compartments and into the pumping compartment.

18.6.3 Return Water Sump Details and Preparatory Works

The key design features of the RWS are summarised in Table 18.6.

Table 18.6 Key Parameters Associated with the Kamoa RWS

Description	Value	Unit
Footprint Area	700	m^2
Storage Capacity	1000	m^3
Depth of Excavation below NGL	3.0	m

18.7 Water Balance

Epoch has not undertaken a water balance study as this is being undertaken by Golder, who are undertaking a site wide water balance for the Kamoa 2015 PFS. A more detailed water balance should be performed in the next phase of the project.

18.7.1.1 Closure Activities at Cessation of Operations

At the cessation of operation of the TSF, the focus will be on the cover and vegetation of the top surface of the facility, the decommissioning of facilities associated with the TSF and the construction of storm water 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 storm water 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 minimize 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 stripping of sufficient 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; and
- 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.

The nature of the available soils likely to be stripped from the footprint of the TSF requires that they are protected against erosion. This will be done by a combination of mixing with selected waste material and the establishment of vegetation to the cover. The mixing of soil with material of a gravel/rocky nature has been found to be effective in improving the erosion resistance of cover layers to sloped areas. The establishment of vegetation to the side slopes of the facility could be done by hand planting, seeding or hydro-seeding and should comprise a mixture of grass, shrubs and trees. The most effective method of covering and vegetation establishment will be arrived at during the operational life of the facility by a process of trial and error. The vegetation used in the establishment of the vegetative cover will all be indigenous and should not require irrigation.

18.8 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;
- The P & Gs supplied by MCK are low, particularly when considering the protracted construction time (15 years);
- A suitable borrow pit has not been identified as of yet for the use in the impoundment wall;
- Geotechnical tests have not been performed on the 100% passing 20 µm tailings sample, which may reduce the permeability and strength parameters; and
- Rehabilitation of the side slopes of the TSF may need to be scheduled earlier, as the impoundment wall will be completed at year 15.

18.9 Opportunities

The possible opportunities associated with the current TSF design are as follows:

- It may be possible to convert the TSF to an upstream facility once the Rate of Rise has reduced sufficiently and stability modelling has been performed;
- Determining what the tailings beach slope will be in reality will give insight into the possibility of running a floating barge system, which may reduce costs.

18.10 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. 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.;
- Review P & Gs as they are low considering the protracted construction time of 15 years; and
- 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.

18.11 Communications

Current site communications comprise satellite phones and a Very Small Apeture Terminal (VSAT) internet link. Cell phone service is available on site as well as in Kolwezi. This will be used for construction and operations. Increased bandwidth and coverage across the whole site will be provided.

18.12 Waste Management

Currently land fill sites or waste collection facilities in the Kolwezi area are limited. So are hazardous waste management contractors or services based in Kolwezi that can deal with oils, batteries, bio hazardous waste etc.

An integrated approach to waste management for Kamoa 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 prefeasibility phase.

18.13 Roads and Earthworks

18.13.1 Main access road

A reliable and safe main access road from Kolwezi to the Kamoa project will be required. A reliable and safe main access road from Kolwezi to the Kamoa project is required. The proposed new and upgraded road sections are shown in Figure 18.14.

Figure 18.14 Proposed Access to Kamoa Site

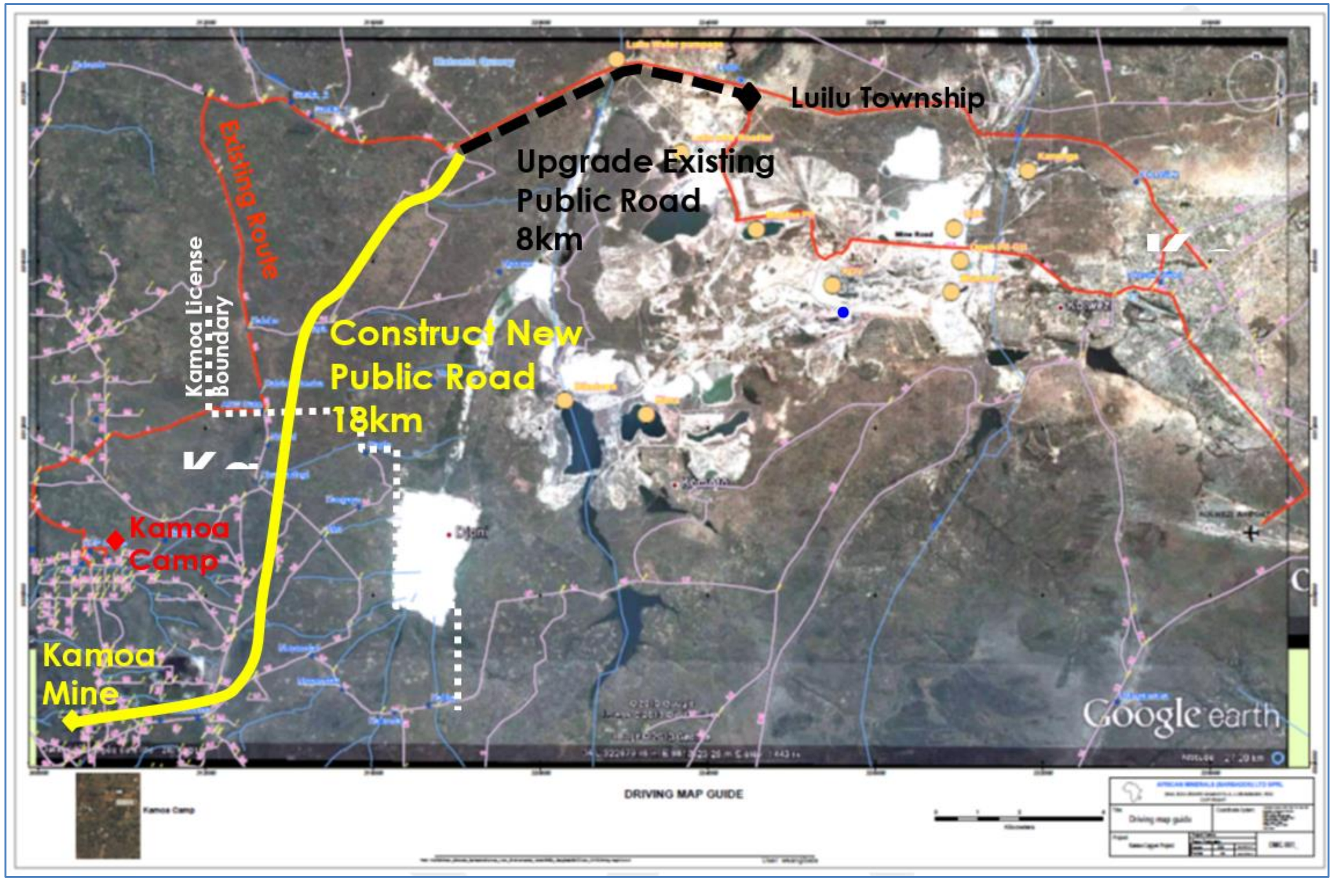


Figure by Ivanhoe, 2016.

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 18.15.

Figure 18.15 Proposed Access Road Construction

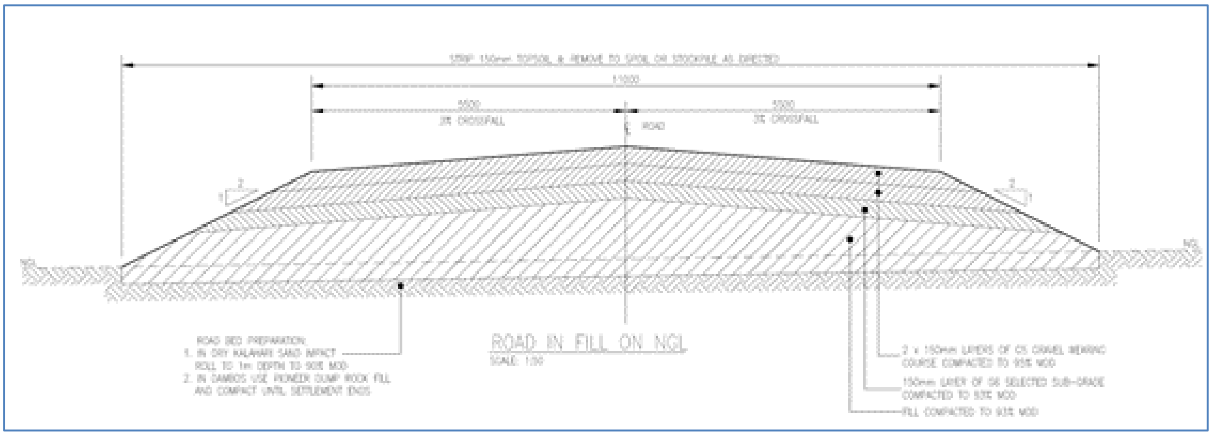


Figure Courtesy of MDM 2015.

There is potential to use this base and surface the road later in the project.

Alternative 1 is approximately 7 km longer than alternative 2. The length of sections to be upgraded are however very similar for alternatives 1 and 2.

From a mine accessibility point of view, alternative 2 is the recommended route. However, land ownership, environmental and social issues are more problematic on alternative 2. For both alternatives new road servitudes will have to be obtained. This servitude will be finalized during future phases of the project.

It is proposed that the access route to the Kamoa Mine should be a bitumen surfaced road within a 30 m minimum road reserve. Typically the road will be designed for a design 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 roads will have a surface width of 9 m with two 3.5 m bitumen surfaced lanes and 1.0 m surfaced shoulders. An additional 1.5 m gravel shoulder has been added to both sides for a total formation width of 12 m. The inclusion of gravel shoulders is mainly for the recovery area if a vehicle breaks down during the prefeasibility phase, Alternative 1 will be developed further to a preliminary design status.

18.13.2 Other Roads

The following facilities have been allowed for inside the plant and mine area:

- Plant roads. All plant roads will be surfaced.
- 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. Variant road widths will be provided, depending on the hierarchy of the road in the village. All roads will initially be gravel roads.

18.13.3 Terracing and Earthworks

The terrace 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 Kamoa site has been identified to consist of collapsible soils of low bearing capacity that shall prove inadequate to support heavy structural foundation loads. The 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.

18.14 Logistics

A phased logistics solution is proposed in the Kamoa 2016 PFS. Initially the North South corridor between southern DRC and Durban in South Africa is viewed as the most attractive and reliable export corridor. As soon as the rail link between Kolwezi and Dilolo is sufficiently rehabilitated, Kamoa will be rail linked to Lobito in Angola, which will then be the closest seaport. It has been assumed that the rail link will be available from Year 2 when this route then becomes the preferred export corridor.

Nonetheless a number of competing 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.

In the initial phase, export using a combination of road and rail along the North–South Corridor through Durban in South Africa is proposed. 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 Chingola and loaded onto trains of 38 wagons with 2 x 20 ft containers per wagon for railing to Durban along the North South corridor.

After a recent major upgrade project, the port of Lobito in Angola has sufficient container handling capacity to serve Kamoa. Similarly the Benguela railway line between Lobito and Luau /Dilolo at the Angola/DRC border has recently been upgraded to a capacity of up to 20 Mtpa. As soon as the rail link between Kolwezi and Dilolo in the DRC is sufficiently rehabilitated, Kamoa will be rail linked to Lobito in Angola, which will then be the closest seaport. The existing rail line between Kolwezi and Dilolo is handling light passengers trains but is in a poor condition and will require upgrading to enable the handling of any significant freight volume. Once upgraded and operational, concentrate will then be bagged and containerized at the mine and railed to Lobito for shipping. Containerizing at the mine is proposed instead of open wagon transport and containerizing at the port of Lobito.

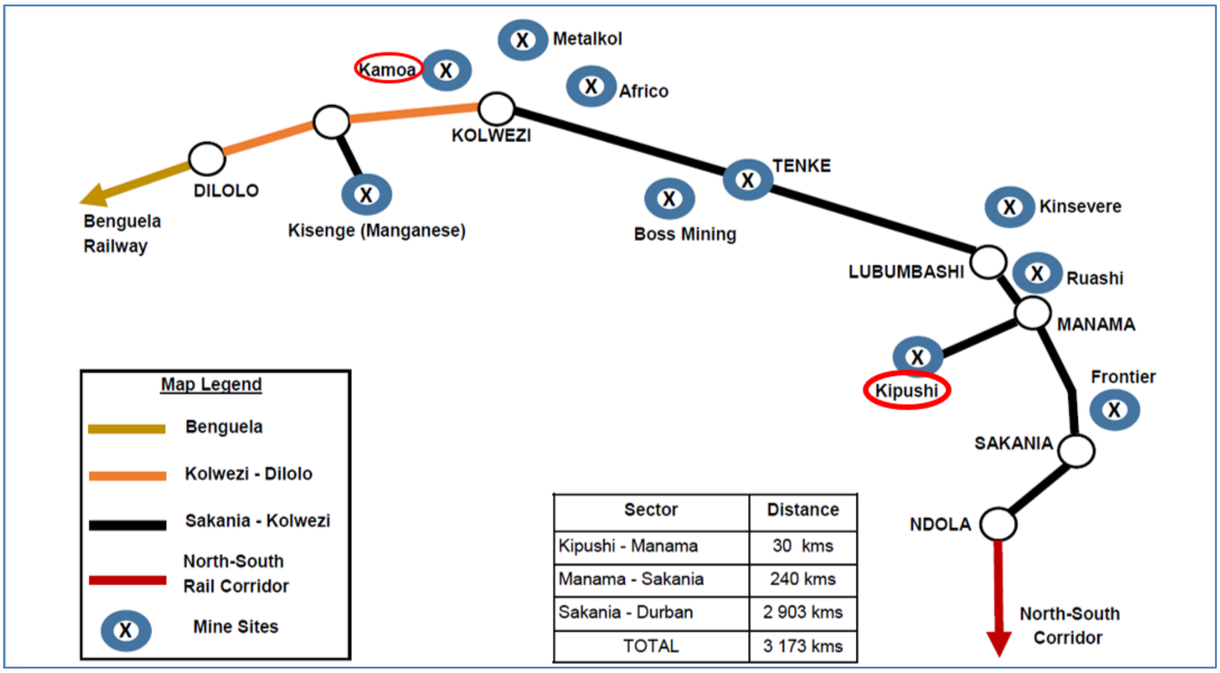
The western rail corridor to Lobito and the north-south corridor through to Zambia is shown as a plan in Figure 18.16 and diagrammatically in Figure 18.17. The North South Corridor is shown in diagrammatically in Figure 18.18.

The use of an operational line between Kolwezi and Lobito port is not exclusively dependant on the rehabilitation of the rail infrastructure; however, it needs joint agreement from both countries' respective governments in addition to completing an institutional framework that should govern these operations.

Figure 18.16 Kamoa to Lobito Rail System

Figure by Grindrod 2015.

Figure 18.17 Western Rail Corridor



Sector	Distance
Kipushi - Manama	30 kms
Manama - Sakania	240 kms
Sakania - Durban	2 903 kms
TOTAL	3 173 kms

Figure by Grindrod 2015.

Figure 18.18 DRC to South Africa North South Rail Corridor

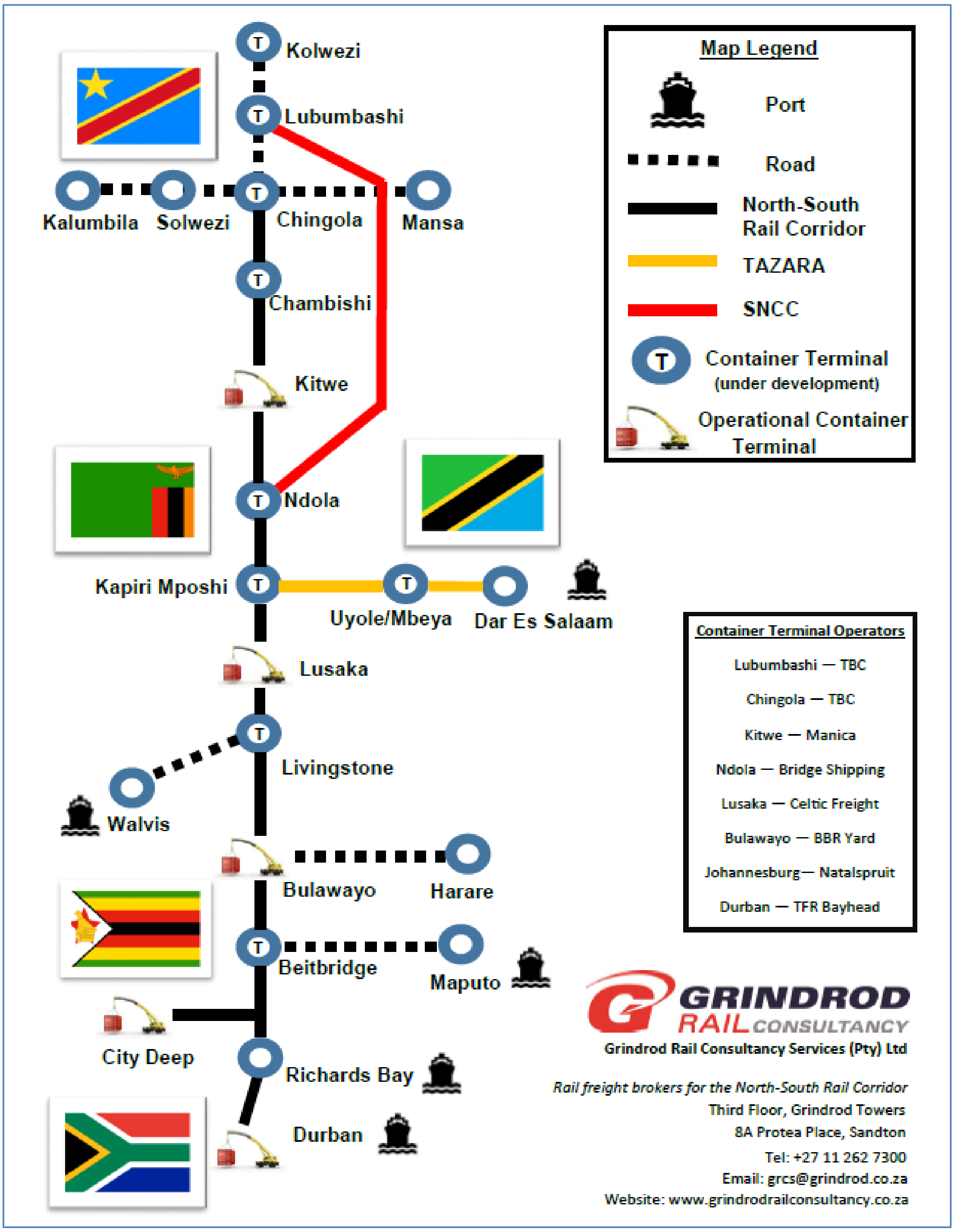


Figure by Grindrod 2015.

18.15 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 ITAB, and a number of smaller airlines and private charters.

The two airports will be utilised to transport people, goods and material to the project site during construction and operations phases.

18.16 Consumables and Services

18.16.1 Fuel

Transport fuel and fuelling infrastructure is available along all of the required routes, albeit fuel quality and standards between countries is likely to vary. This may prove a problem in future, particularly if certain countries impose higher emission standards (EURO VI) on vehicles and the quality of fuel required to meet these standards is not available in each of the countries transited.

18.16.2 Maintenance

There are no commercial type break down facilities between Chingola, Zambia and Lubumbashi, however, there are commercially-owned breakdown rigs with a towing capacity of up to 30 t.

18.16.3 Power Lines

Power lines crossing roads are evident in places and the impact thereof on the transport of abnormal loads needs to be investigated.

18.16.4 Bridges

A preliminary bridge assessment has been undertaken. However, a more in-depth investigation is required particularly with respect to abnormal loads. This will be undertaken during subsequent project development phases. The Kapolowe Bridge (30 km from Likasi) and the bridge at the Lualaba River (35 km from Kolwezi) in particular require a more in-depth investigation with respect to abnormal loads.

18.16.5 Weather

The impact of weather on the movement of vehicles over river crossing and dirt roads has not been ascertained. This will be undertaken during subsequent project development phases.

18.16.6 Logistics Companies for 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.

18.16.7 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.

The routing of reagents and consumables to Kamoa will be the subject of a future, separate transport study.

18.16.8 Schedule

From the above, it is evident that the movement of goods and materials to and from site will require extensive planning and monitoring to ensure on time arrival.

18.17 Water and Wastewater Systems

18.17.1 Water Demand

The estimated water demand for the project scenario is given in Table 18.7.

Table 18.7 Estimated Water Demand

Description	Units	Quantity
Mining Water Requirement	m^3/day	160
Concentrator Water Requirement	m^3/day	4,800
Potable Water Requirement	m^3/day	140
Total Daily Requirement	m^3/day	5,100
Total Daily Requirement	Mℓ/day	5.1

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. A return water pipeline (10 km) will bring water from the TSF to the process water tank.

18.17.2 Bulk Water

The assessment of the bulk water supplies has been undertaken with the view of supplying the estimated water demand of 5.1 ML/d.

Two potential sources have been identified for the bulk water supply:

- The aquifer developed within the sandstone forming the Kamoa and Makalu Domes and the footwall to the mining operations, and
- 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.

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.

The Haute Luilu Dam for water supply is considered as a long term contingency at this stage. 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.

18.17.3 Potable Water

Potable water for local villages is currently obtained from local rivers and streams. All watercourses occurring within a 5 km radius of a village are considered extraction points for potable water. In most villages, hand-dug wells supplement domestic water supply when natural watercourses dry up. Potable water for any future mining operation will be sourced from bore holes. The Lufupa River, located to the south–west of the Project, is also a potential water source, but has not been investigated.

Potable water for the mining operation will be obtained from groundwater. 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.

Potable water will be distributed via pipe racks and sleeper ways along with other services where possible and underground as necessary.

18.17.4 Storm Water Infrastructure

The Department in Charge of the Protection of the Mining Environment in the Democratic Republic of Congo (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 storm water management plan, the pollution control dams and the pipelines with their required pumps are all based on these regulations.

18.17.4.1 Storm Water 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 1 in 100 year return period 24 hour rainfall event.
- The storm water management plan and pipeline system was developed based on the most current site arrangement information available to Golder Associates Africa.

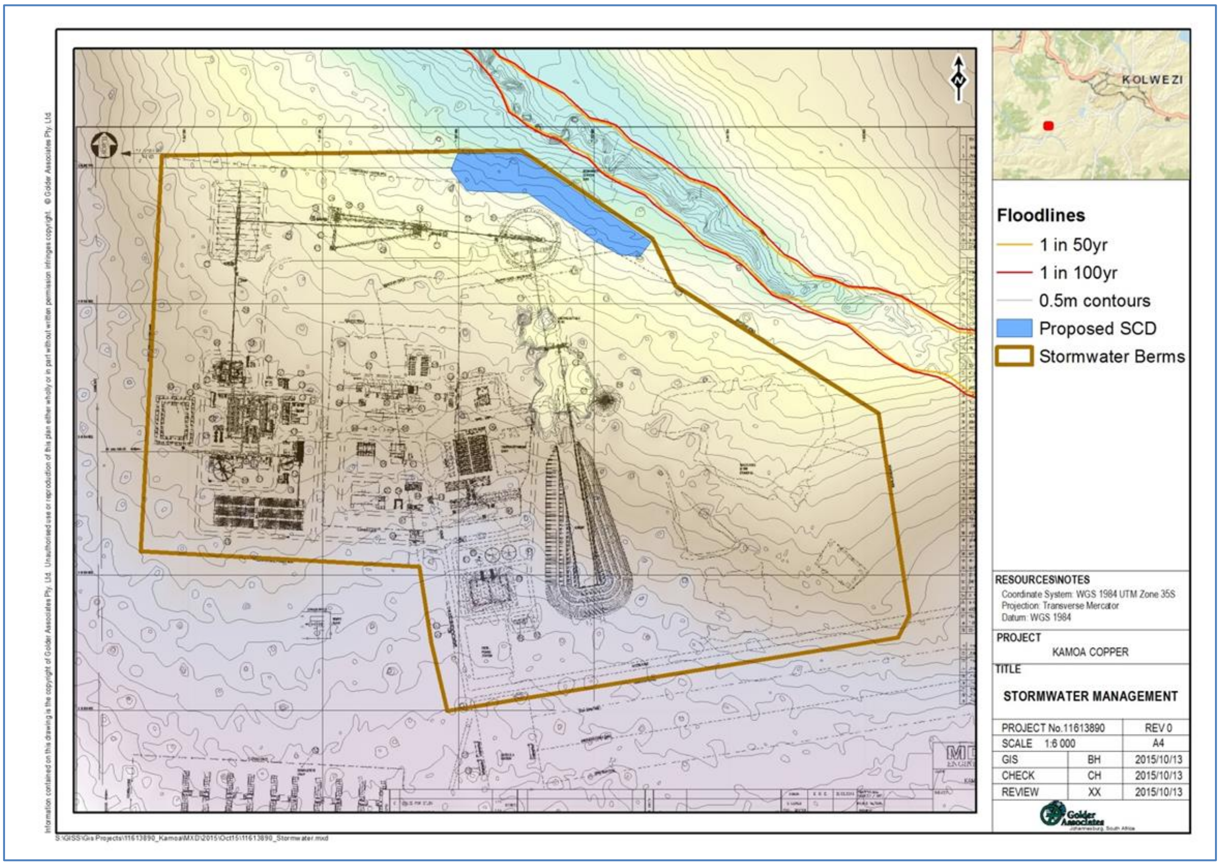
The location of the plant, stockpile and decline area is shown in Figure 18.10. The run-off from this area will be polluted and will need to be managed within the mine's dirty water system. Berms are required around the perimeter of the area to prevent run-off from the upslope areas entering the site. 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 1 in 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 recommended at prefeasibility stage.

A higher detailed storm water management plan and pipeline system will be developed as the mining project progresses. The storm water management plan included in this document is done at a high level and should be considered a conceptual plan.

Figure 18.19 Site Layout



18.17.5 Wastewater

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. Floor washings that are potentially contaminated with mineral oils (workshops, refuelling and lube and diesel storage areas) will drain to the run-off dam.

Sludge from the sewage plant will be pumped out with tailings or dried on constructed drying beds and buried.

Other wastewater streams and by-products such as acid are covered under plant process design.

18.17.6 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 acid. 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 other heavy metals.

However, as the water balance turn 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 to address acidity, presence of metals and salts in the mine water will amount to approximately US$1 M for a Water Treatment Plant with a capacity to treat 1 ML of water/day.

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, in the event that water treatment would be required beyond closure.

18.18 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 overall system will include a combination of passive measures (e.g. fire walls, physical isolation etc) and active systems (eg. 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 would be capable of providing 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.

18.19 Hospital and Medical Facilities

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-patriot employees will be provided by an outside contracting service.

18.20 Mine and Concentrator Building Requirements

The buildings that will form part of the project infrastructure are listed below.

18.20.1 Single Story Brick Buildings

- Administration building/offices.
- Clinic and first aid station.
- Tea room/canteen.
- Plant security office.
- Change house.
- Control room.
- Laboratory.
- Gate house & security.
- Laundry.
- Training.
- Mess Complex
- Satellite ablutions.
- Substations

18.20.2 Structural Steel Buildings

- Plant and Mine workshop (light crane loads).
- Plant store.

18.20.3 Additional Building

- Explosive Storage

18.21 Owner's Camp

18.21.1 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 units with be 3 bed, two bathroom with open plan living room and kitchen.

18.21.2 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.

18.21.3 Roads and Services

The following roads and services will be provided in the accommodation area:

- Perimeter security fence.
- Paved roads designed to an appropriate residential standard around the town centre.
- Gravel access roads to housing units.
- Parking (remote from rooms).
- Stormwater drainage.
- Water reticulation, sized for fire flows and provided with hydrants.
- Sewer reticulation.
- Sewage treatment.
- Street lighting.
- Transmitters and receivers for mobile telephones.
- Internal and external communications.

18.22 Construction Facilities

To facilitate the execution of the project, various temporary facilities need to be put in place. 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 operations. The camp plan assumes single 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 pay TV will be provided. As the camp will be used during the operational phase for junior and on-shift accommodation it will be built for a 25 year service life
- Construction Site Offices: Site offices will be needed for 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 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 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.

18.23 Comments on Section 18

Infrastructure planning is at a relatively high level for a PFS and no issues were identified that will have a material negative impact upon the financial viability of the project.

19 MARKET STUDIES AND CONTRACTS

The Kamoa 2016 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 analysisfor 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 2016 PFS assumes a copper price of $3.00/lb, this is consistent with long term estimates and pricing used in other published studies.

There is potential to sell copper concentrate to smelters in Zambia and or merchants where more favourable terms may be possibleThe 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 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 fedd by copper concentrate feed from the mine.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Studies and Issues

20.1.1 Background

The Kamoa Copper Project (the Project) is 25 km west of Kolwezi, and is located within the newly formed Lualaba Province (this Province was formed out of Katanga in July 2015), Kolwezi district, Mutshatsha Territory in the Luilu and Lufupa Sectors, and the areas of Chief Musokantanda (in the south) and Chief Mwilu (in the north), see Figure 20.1. The mine licence is located in a rural area with no mining or industrial activities. Extensive urbanisation, industry, and mining occur in Kolwezi 25 km to the east.

The Project is primarily contained within Exploitation Permits 12873, 13025, and 13026 (formerly part of Exploration Permits 702, 703, and 705), but encompasses smaller portions of the areas of the additional one Exploration Permit 703. The Project has approximate dimensions of 45 km north–south by 20 km east–west, and covers an area of approximately 400 km² (Figure 20.2). The project is in an early phase of construction with the initial boxcut completed and declines planned for development commencing the third quarter 2015. Construction of the broader 3 million tonnes per annum (mtpa) project will commence following authorisation of the updated Environmental Impact Study (EIS) due for completion during the third quarter of 2016.

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 is often associated with HIV/AIDS infections, and malnutrition.

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 Kamoa 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).

Figure 20.1 Project Locality

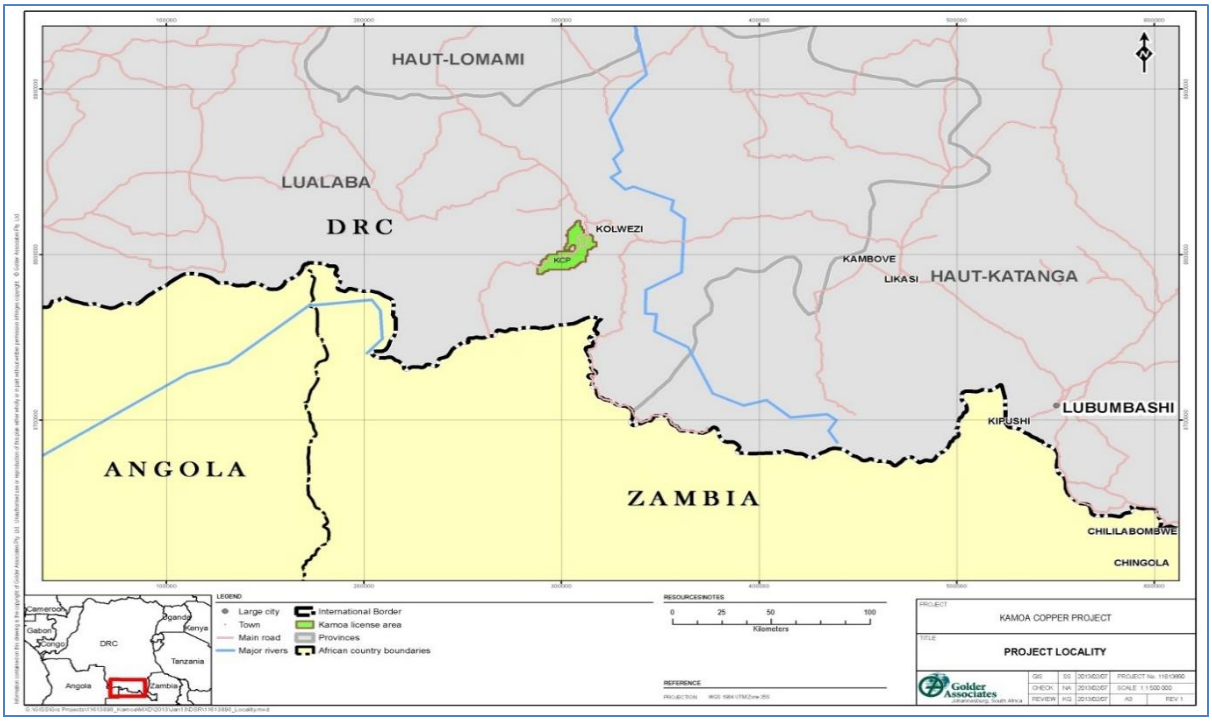
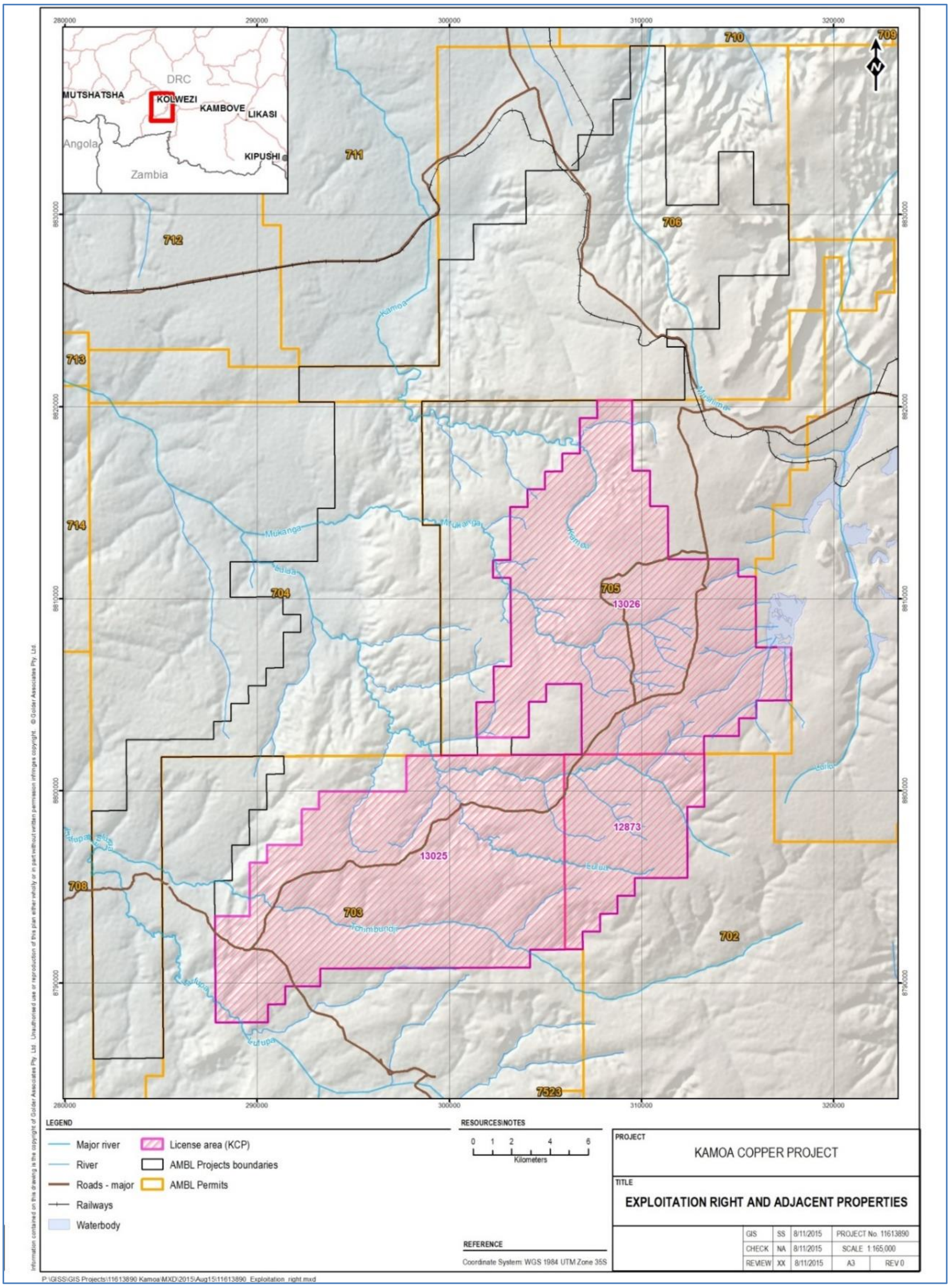


Figure 20.2 Kamoa Copper Project Exploitation Right and Adjacent Properties



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 above sea mean level (mamsl). Current exploration activities are at elevations ranging from 1,450 to 1,540 masl. The local topography of the Project is affected by the drainage catchments of the Mukanga, Kamoa and Lulua Rivers and the Kalundu, Kansoko and Kabulo Streams (Figure 20.3). 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 northeast from Angola including the southeast 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 November to March. The woodland is interspersed with Dambos, (grassy wetlands), which may constitute up to thirty per cent 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 20.4).

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 as per Figure 20.5. This EIS was based on conceptual planning information. This has subsequently been updated through the ongoing feasibility 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 feasibility 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 Kamoa's intention 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; and
- Ongoing community, interested and affected party as well as Government consultation.

Figure 20.3 Surface Water Catchments

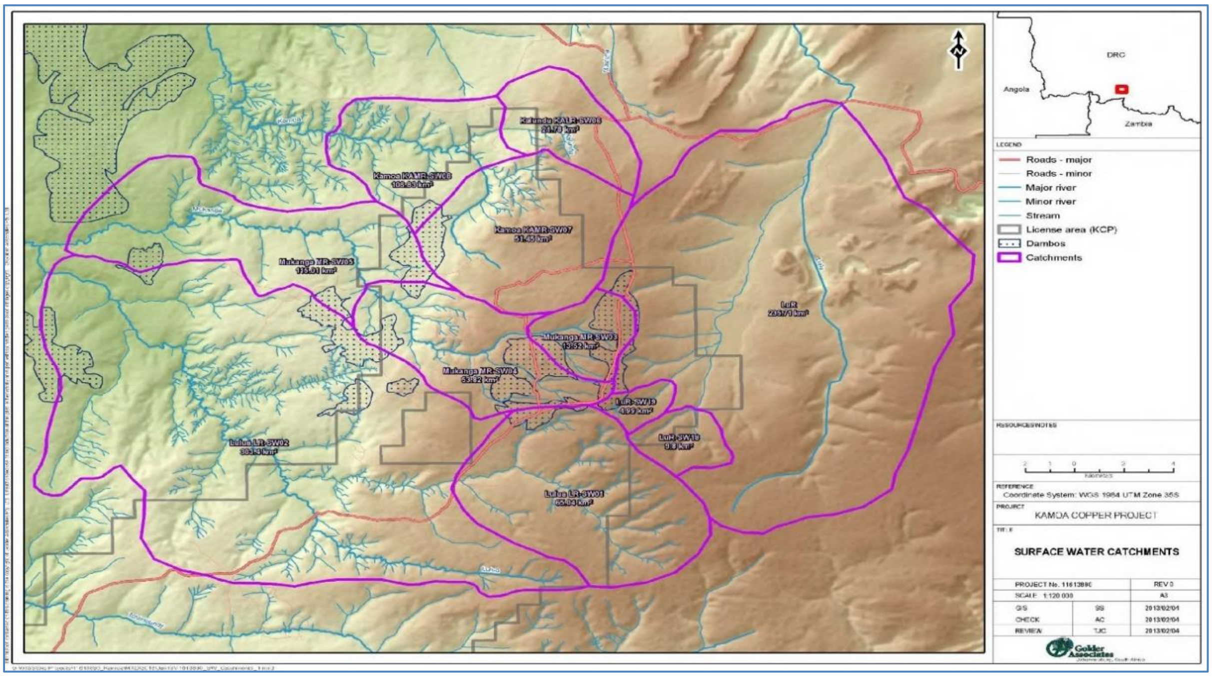


Figure 20.4 Environmental Map of the Project Area

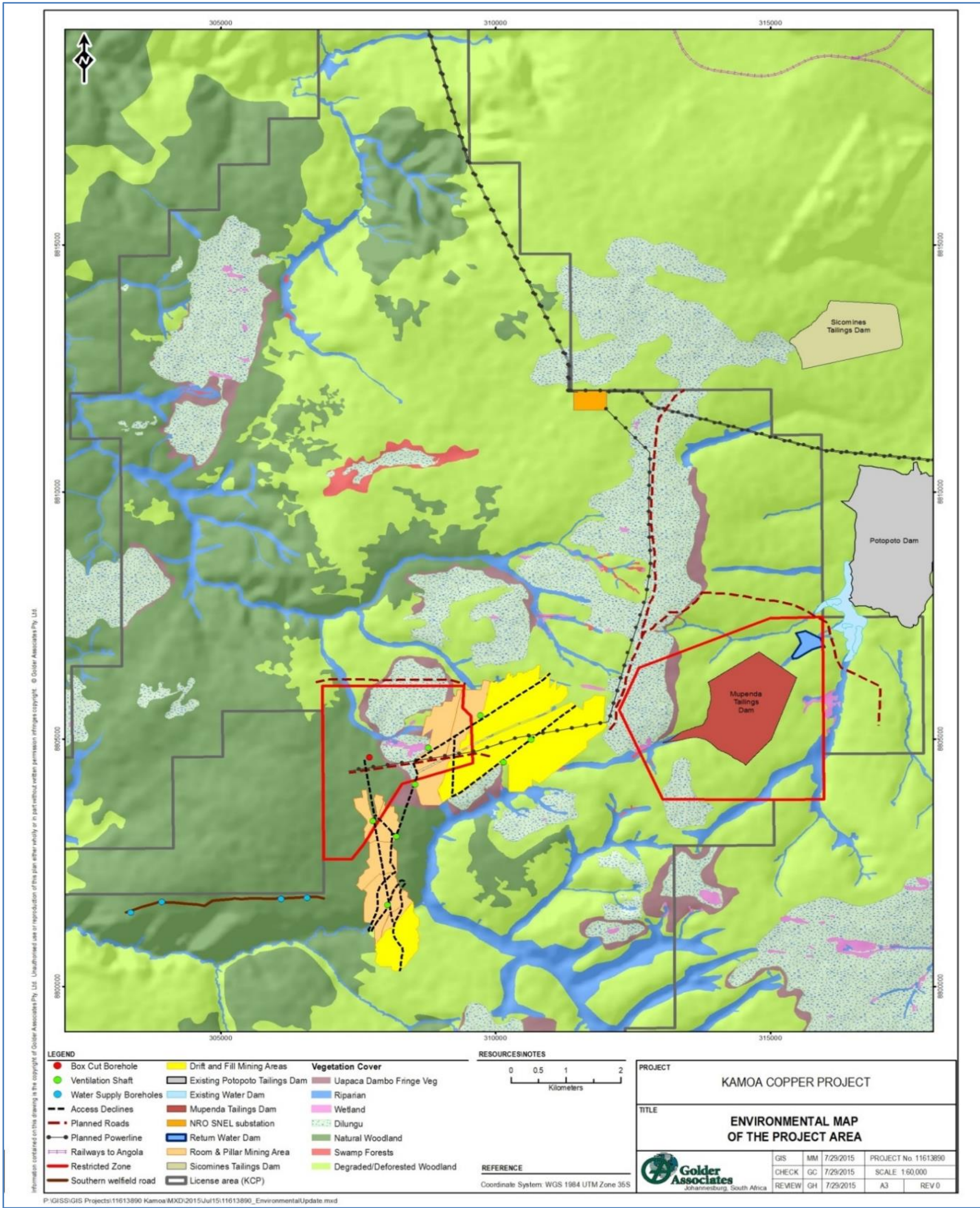
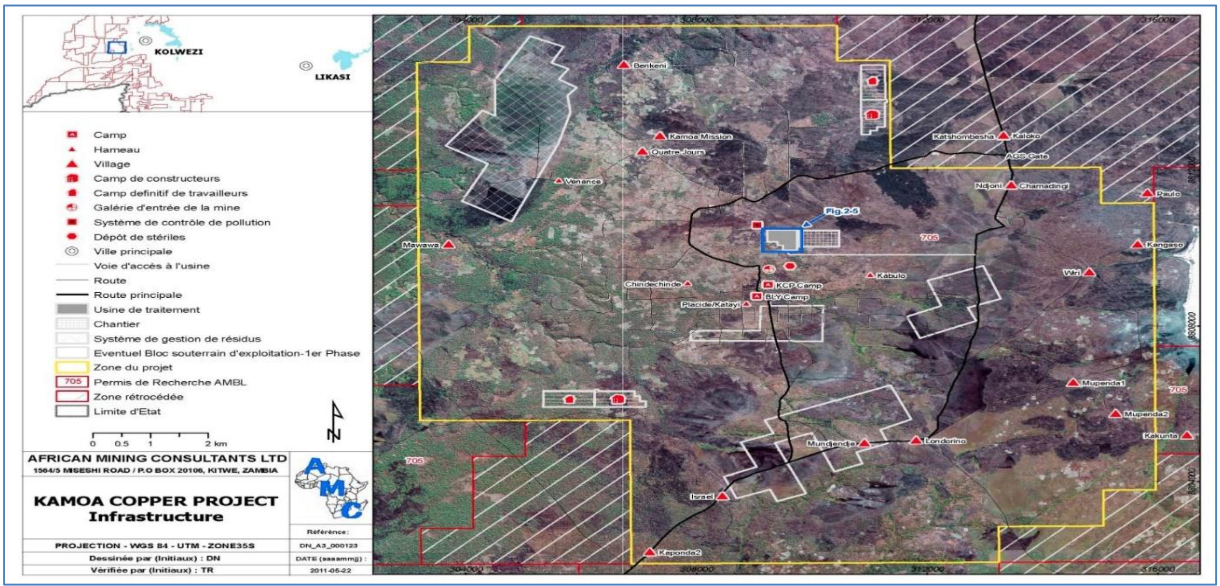


Figure 20.5 Approved Mine Plan as per EIS (African Mining Consultants, 2011)



The above 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 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. The EIS update will likely recommence 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, and will be based on the definition of Phase 1 as per the PFS (3 Mtpa underground mine with concentrator plant). Resubmission to Government is estimated for the fourth quarter of 2016. 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 will be included in the EIS update.

Third party developments in support of the Project include the development of a power line from the national grid to the plant and various upgrades to existing hydroelectric schemes. In March 2014, a financing agreement was signed between Ivanhoe and the DRC's national electricity company, La Société Nationale d'Electricité (SNEL). Ivanhoe is working with SNEL to upgrade two existing hydroelectric power plants – Mwadingusha and Koni – to generate up to 113 megawatts of power to be made available to the national supply grid. SNEL will provide the Project with up to 100 megawatts from the grid, which would be sufficient to operate the initial phase of the Kamoa mine. A third hydroelectric power plant – Nzilo 1 – would follow under the same financing agreement. Nzilo 1 will have a generating capacity of approximately 108 megawatts. This will entitle Kamoa to an additional 100 megawatts from the grid. The technology planned to be used for power plant upgrades will increase the original design capacity of these plants by up to 10%.

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; and
- 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 Kamoa's ability to extract the Mineral Resources or mineral reserves relating to current development/operations were determined as follows:

- 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 (1 week to 1 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 (Figure 20.6):

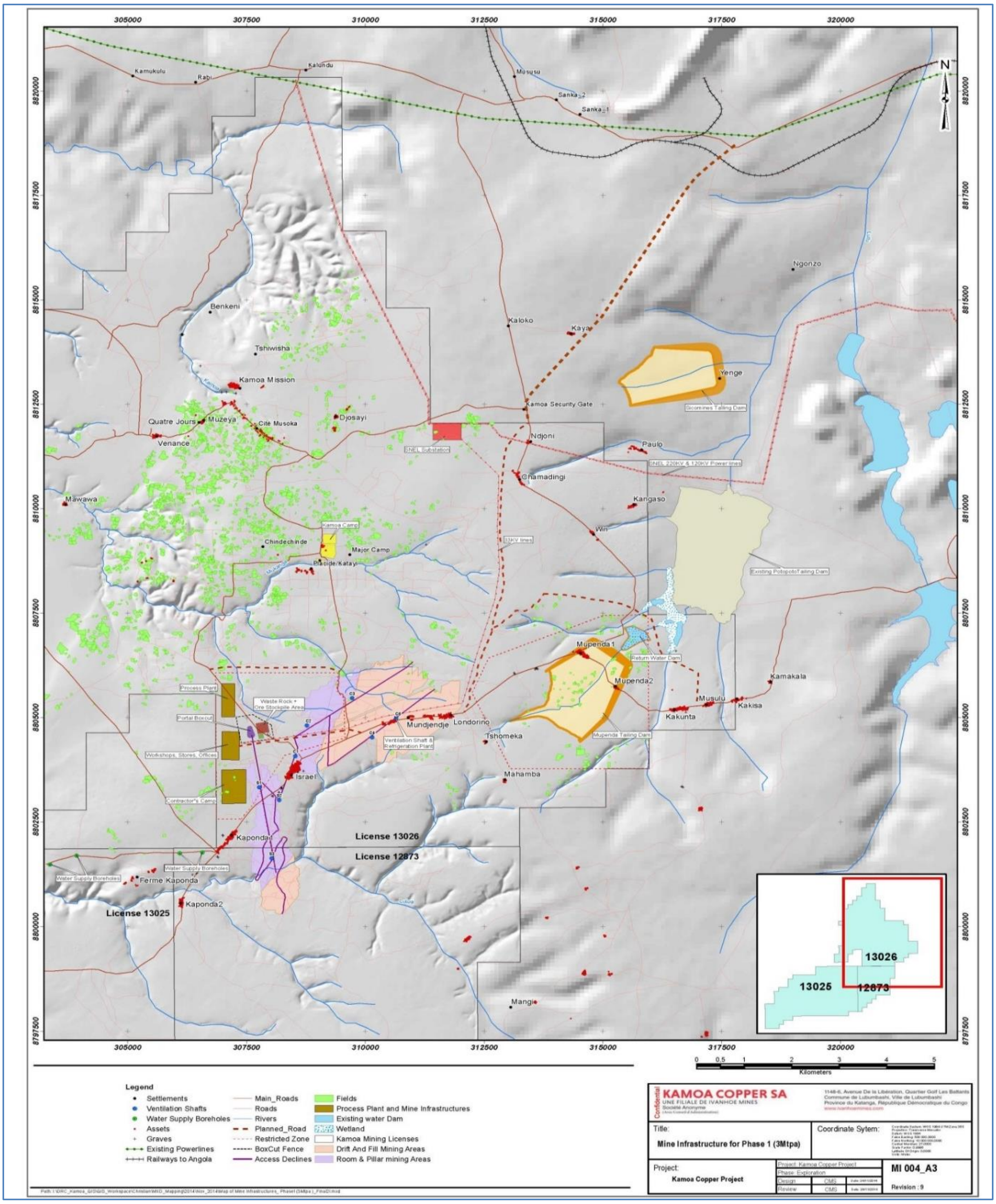
- A number of options have been considered for the decline development to ensure that the workings remain dry, including the construction of dewatering boreholes, sealing decline with grouting and others. Dewatering of the decline from dewatering boreholes planned to be drilled in the second quarter 2016. 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 at Mupenda, (Figure 20.6) 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; and
- Resettlement and compensation of persons who may be affected by physical and economic displacement (e.g. Mupenda 1 and Mupenda 2 for the tailings location, Figure 20.6). 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 PFS phase and EIA modelling to determine resettlement based on impacts. Terms of References for the RAP were prepared in August 2014.

The EIS update and RAP would need to be approved and the RAP implemented prior to Project construction scheduled for the third quarter of 2016. For this reason Kamoa intends to commence with these studies in the first quarter of 2016 for approval and implementation of the RAP by the third quarter of 2016. These deadlines are tight.

Figure 20.6 Latest Mine Plan (Kamoa Copper SA July 2015)



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, discussion 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); and
- To ensure that waste management at Kamoa is conducted in a legally compliant manner.

20.2.2 Tailings Management and Disposal

No tailings are currently being generated by the Project. Once the concentrator is operational (anticipated in 2018 tailings will be produced that will require disposal. Following a number of site selection evaluations by Kamoa, the most favourable site selected for the location of the Tailings Storage Facility (TSF) was the Mupenda site (Figure 20.6). This site is deemed favourable as (Epoch, 2015):

- It is located within an already disturbed catchment due to the failed tailings dam downstream of it;
- The site does not require a liner to be installed over the entire footprint; and
- It is able to cater for additional storage capacity should the mine life be extended or the tailings production rate increased.

The key design features of the TSF (Epoch, 2015) are as follows:

- The TSF will be constructed as a valley impoundment dam with a compacted earth embankment wall; The TSF will be constructed as a downstream facility;
- The wall will be raised in 5 phases, where Phase 1 is at elevation 1468 mamsl and the last phase is at elevation 1492 mamsl;
- The TSF has a total footprint area of 257 ha, a maximum height of 42 m and a final rate of rise of <1 m/year;
- A concrete lined Return Water Sump with a water storage capacity of 1,000 m³;
- Minimal resettlement due to land impacts; approximately 80–100 households will be

physically and/or economically impacted;

- A slurry spigot pipeline along the crest of the TSF; and
- Tailings will be delivered to the TSF through a dedicated pipeline and water recovery through a return water dam for reuse.

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 $\leq 1 \times 10^{-6}$ 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 1×10^{-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

Kamoa's environmental management team comprises eight permanently employed staff members working within the five environmental management pillars shown in Figure 20.7.

Figure 20.7 Environmental Management Pillars - Kamoa

20.2.4 Site Monitoring

Environmental, social and health baseline data collection and ongoing monitoring has been carried out within 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 (NO2), sulphur dioxide (SO2) and ozone (O3). Subsequent to the initial monitoring campaigns, Kamoa 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 boxcut area and Israel village during the blasting campaigns in 2015.
- Soils, Land Use and Land Capability – two 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. Ecological monitoring is undertaken on an ongoing basis at Kamoa. 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 minimization 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 EIA purposes.
- Health – specific health information was collected for the Project in 2013 by Golder for EIA purposes; and
- Archaeology – archaeological and cultural heritage surveys of proposed infrastructure development areas were undertaken in 2011 for the initial EIA and 2013 for the update of the EIA by Golder.

20.2.5 Water Management

The water demand for Kamoa Copper is estimated to be 5.1 ML/d for the planned 3 Mtpa mine. 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 kilometre's to the SW of the mine site on the southern portion of the Makalu Dome, (Golder 2015). The position of the 4 production boreholes is depicted on Figure 20.6. 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 1 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); and
- The Ministry of Mines issued three Exploitation Permits to Kamoa on 21 August 2012.

20.3.1.1 Financial Guarantee

In terms of the financial guarantee required by DRC Law, the EIS included an estimated of the total closure costs, amounting to US$8.1 M. The payments as financial guarantees are listed in Table 20.1.

Table 20.1 Payment tranches for the Financial Guarantee

Year	Year (US$ 000)
2013	–
2014	31
2015	91
2016	153
2017	214
2018	275
2019	337
2020	–

In terms of the annual financial guarantee payments as required by the DRC Mining code, Kamoa has made the required payments for 2014 and 2015 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

As per the DRC mining code, the developer must present a sustainable development plan (SDP) as part of the EIS. The SDP presented in the EIS (African Mining Consultants, 2011) and approved by DPEM commits Kamoa to empower local communities and improve their welfare as follows:

- Skills development – to equip the local workforce with skills for the mining and industrial sector;

- Commercial development – assistance with the establishment of local goods and service enterprises to supply the Project and wider mining sector;
- Social and cultural development – aiming at improving community infrastructure and services as guided by the local communities themselves through established community development committees;
- Sustainable livelihood projects - From 2010 to mid-2014, Kamoa hired Eco-livelihoods UK Ltd. to implement a Sustainable Livelihoods Project in the Kamoa area, the project was taken over by Kamoa in mid-2014. Part of the Project's objectives is to address food security in the area through an extension programme introducing Conservation Agriculture (CA) techniques, systems and cultivars; in addition the provincial government requires the establishment of 500 ha of crops by mining companies. Demonstration plots have been established and lead farmers trained with the creation of CA plots in the community. This has been rolled out throughout the community with good results being achieved in improving yield, variety and resilience; and
- Local employment and economic opportunities – Kamoa has implemented a policy that looks to place preference to local communities for employment. More than 60% of Kamoa's staff compliment is from local communities within a 15 km radius of the Project.

In terms of its obligations to the regulator, Kamoa tracks its spending and progress against the SDP on a monthly basis and provides annual reports to the regulator relating to the SDP. Inspections by DPEM (Division for the Protection of Mining Environment) are regularly conducted in order to confirm Kamoa's adherence to SDP and Environment Management Plan. The SDP will be further developed during the EIS update adopting international best practices for strategic community investment. Kamoa currently employs 3 Community relations staff and five livelihood staff. The pillars of the social team are:

Figure 20.8 Social Management Pillars – Kamoa

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 Kamoa's ability to extract the reserves or resources. It should be noted that upcoming local and national elections in DRC in late 2016 early 2017 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 20.2).

Table 20.2 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 Ivanhoe policies Local Development Plans
Employees : Strikes, sabotage	Reputation, relationship with government deteriorates, financial loss, project delays.	Human Resources strategy Local hiring procedure Union relationship
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
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 mobilization, unsustainable dependency, reputational issues	Local Development Plans Regional Development

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 & Commitment - Sustainability policy, Environmental and Social responsibilities and accountabilities;
- Hazard Identification & Risk Management - Risk Assessments and Risk Register.
- Legal & Other Requirements - Register of legal obligations. Sustainability objectives and targets, Social and Environmental Improvement Plans;
- Objectives Targets & Performance Management – specific indicators, reporting parameters;
- Training Awareness & 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 & 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 & 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 & Grievances Reporting & Management registers - Records of incidents and grievances management (e.g. investigation reports and filled grievances forms); and
- Monitoring Audits & 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.

The phase 1 PFS currently defines a 24 year mining plan; however the resource is sufficiently large to support multiple expansion phases that could extend the life of the mine well beyond 24 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; and
- 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 board and pillar underground mining methods will be followed, surface subsidence is possible if secondary pillar extraction is undertaken (currently not foreseen). However, as part of mining operations backfilling of the workings will be conducted. 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 boxcut for the decline to the planned underground workings. A network of dewatering boreholes may be established to dewater the boxcut 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

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 as a per tonne basis.

Table 21.1 Unit Operating Costs

	US$/lb Contained Cu in Concentrate		
	Total Average	**Years 1–5 Average**	**Years 6–10 Average**
Mine Site Cash Cost	0.75	0.55	0.75
Realisation	0.73	0.72	0.74
Total Cash Costs	**1.48**	**1.27**	**1.48**

Table 21.2 Operating Costs

Description	Total	Years 1–5 Average	Years 6–10 Average	Total Average
	US$M	US$/t Ore Milled		
Site Operating Costs				
UG Mining	2,453	24.6	37.1	34.1
Processing	886	12.4	12.3	12.3
Tailings	25	0.4	0.4	0.4
General & Administration	511	8.1	7.1	7.1
SNEL Discount	-109	-1.5	-1.5	-1.5
Customs	60	0.7	0.9	0.8
Total	**3,826**	**44.6**	**56.2**	**53.2**

The capital costs for the project are summarised in Table 21.3.

Table 21.3 Capital Cost Summary

Description	Initial Capital US$M	Sustaining US$M	Total US$M
Mining			
Underground Mining	508	499	1,007
Subtotal	**508**	**499**	**1,007**
Power			
Power Infrastructure	19	4	23
Power Supply Off Site	104	–	104
Subtotal	**123**	**4**	**127**
Concentrate & Tailings			
Process Plant	160	8	167
Tailings	23	62	84
Subtotal	**182**	**69**	**252**
Plant Infrastructure			
Plant Infrastructure	18	4	23
Other Infrastructure	8	2	10
Owners Camp	10	2	12
Contractors Camp	23	5	28
Subtotal	**59**	**14**	**73**
Indirects			
EPCM	58	–	58
Subtotal	**58**	**–**	**58**
Owners Cost			
Owners Cost	95	–	95
Closure	–	67	67
Subtotal	**95**	**67**	**162**
Capital Expenditure Before Contingency	**1,025**	**655**	**1,679**
Contingency	189	44	233
Capital Expenditure After Contingency	**1,213**	**699**	**1,912**

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 24 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 2015 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.

The underground capital costs were estimated for the following costs:

- Portal
- Underground Development – cost estimate includes eguipment purchase and maintenance, labor and material consumed.
- Surface Buildings and Electrical
- Cover Drilling and Exploration Drilling
- Paste Fill Pipe Drilling
- Paste Fill Plant
- Underground Infrastructures, Electrical and Instrumentation
- Piping Services and Water Handling
- Vehicles – Light vehicles and utility vehicles
- Ventilation and Refrigeration
- Mine Managment 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, drift-and-fill stoping, and paste backfill prepataion, reticulation and placement. Annual operating costs were generated based on the tonnes produced each year.

The underground operating costs were estimated for the following costs:

- Production Direct Costs
- Paste Backfill
- Materials Handling
- Engineering / Mining Stores
- Training
- Indirect Operating Costs - include pumping, ventilation, compressed air, mine service crews, mechanics and electricians for mine operations, and mine site staff personnel.
- Power Costs
- Undefined Allowance

A summary of the 24 year mine plan total and average operating costs per tonne of ore is shown in Table 21.4.

Table 21.4 Underground Operating Cost Summary

Description	**Unit Cost (US$/t Ore)**
Production Direct Costs	13.02
Paste Backfill	9.50
Materials Handling	3.89
Engineering / Mining Stores	1.08
Training	0.09
Indirect Operating Costs	3.37
Power Costs	2.00
Undefined Allowance	1.17
Total Operating Cost	**34.12**

21.3 Concentrator Costs

The capital and operational costs for the concentrator were prepared for the Kamoa 2016 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.

The base date for MDM's capital cost estimate for the concentrator is June 2015.

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 & 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 wer 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.

P&Gs 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 and C&I items 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 ton 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 undrerground 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 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 & Gs) 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; and
  - 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 opportunites 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 Kamoa 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 storm water 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;
- Jointing of Pipes can be done with continuous welding.

21.5.1 Wellfield Development Capital and Operating Costs

Four production boreholes are required to supply the estimated 5.1 ML/d for the 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 production year 1.

21.5.2 Storm Water Management Plan

The following assumptions were made when developing the cost estimates for the storm water management plan:

- South African construction rates were escalated 20% ;
- Petrol and labour are included in the rates;
- No allowances for 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 & General Expenses
- Maintenance
- Equipment & Sundry
- Fuels & Utilities

- Other Offices
- Insurance & Insurance Taxes
- IT Hardware & Software
- Personnel Transport
- Training
- Communications
- Licenses & Land Fees
- Labour Expatriate
- Labour Congolese
- Accommodation & Messing
- Medical Support
- Expatriate Flights
- Light Vehicles
- Environmental
- Community Development
- Banking and Audit Fees
- Legal & 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 the Company'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 2016 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, Kamoa'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$300/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 conceptual 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.

For 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 QP's opinion the work completed adequately supports this level of study estimate.

22 ECONOMIC ANALYSIS

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 facility, and associated infrastructure. The mining rate and concentrator feed capacity is 3 Mtpa. The production scenario schedules 71.9 Mt at 3.86% Cu over 24 years, producing 6.1 Mt of copper concentrate, containing 2,394 ktof 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 Democratic Republic of Congo (DRC) legislation and general industry terms. The economic analysis returns an after tax Net Present Value (NPV) at an 8% discount rate of US$986 M. It has an after tax internal rate of return (IRR) of 17.2% and a payback period of 4.6 years. The life-of-mine average total cash cost after credits is US$1.48/lb of copper.

The key results of the Study are summarised in Table 22.1.

Table 22.1 Study Results Summary

Item	Unit	Total
Ore Processed		
Quantity Plant Feed Treated	kt	71,893
Copper Feed Grade	%	3.86
Concentrate Produced		
Copper Concentrate Produced	kt (dry)	6,106
Copper Concentrate Recovery	%	86.36
Copper Concentrate Grade	%	39.20
Copper In Concentrate	Mlb	5,277
Key Financial Results		
Initial Capital	US$M	1,213
Mine Site Cash Cost	US$/lb Payable Cu	0.75
Realisation	US$/lb Payable Cu	0.73
Total Cash Costs After Credits	US$/lb Payable Cu	1.48
Site Operating Costs	US$/t milled	53.22
After Tax $NPV_{8\%}$	US$M	986
After Tax IRR	%	17.2%
Project Payback Period	Years	4.6

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

		Before Taxation	After Taxation
Net Present Value (US$M)	Undiscounted	5,791	4,096
	4.0%	2,979	2,036
	6.0%	2,152	1,429
	8.0%	1,549	986
	10.0%	1,104	657
	12.0%	768	409
IRR	–	20.7%	17.2%
Project Payback (years)	–	4.1	4.6

Table 22.3 Mine Production and Processing Statistics

Item	Unit	Total LOM	Years 1–5	Years 6–10	LOM AVG
Ore Processed					
Quantity OreTreated	kt	71,893	2,934	3,008	2,996
Copper Feed grade	%	3.86	4.35	4.08	3.86
Concentrate Produced					
Copper Concentrate Produced	kt (dry)	6,106	283	271	254
Copper Recovery	%	86.36	87.06	86.68	86.36
Copper Concentrate Grade	%	39.20	39.20	39.20	39.20
Contained Metal in Concentrate					
Copper	kt	2,394	111	106	100
Copper	Mlb	5,277	245	234	220
Payable Metal					
Copper	kt	2,314	107	103	96
Copper	Mlb	5,102	237	227	213

22.2 Democratic Republic of 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 January 2016. It is presented in 2015 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 payable copper concentrate is 96.68%.

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 (for transport via Durban) for the first two years of production and thereafter US$300/t (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 Democratic Republic of Congo. 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 Democratic Republic of Congo. 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:

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.

Royalties

The royalty is due upon the sale of the product and is calculated at 2% of the received price 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.

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; and
- 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$60/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 the company 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 2016 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 facility, and associated infrastructure. The mining rate and concentrator feed capacity is 3 Mtpa. The production scenario schedules 71.9 Mt at 3.86% Cu over 24 years, producing 6.1 Mt of copper concentrate, containing 2,394 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 Democratic Republic of Congo (DRC) legislation and general industry terms. The economic analysis returns an after tax Net Present Value (NPV) at an 8% discount rate of US$986 M. It has an after tax internal rate of return (IRR) of 17.2% and a payback period of 4.6 years. The life-of-mine average total cash cost after credits is US$1.48/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 2016 PFS Results Summary

Item	Unit	Total
Ore Processed		
Quantity Plant Feed Treated	kt	71,893
Copper Feed Grade	%	3.86
Concentrate Produced		
Copper Concentrate Produced	kt (dry)	6,106
Copper Recovery	%	86.36
Copper Concentrate Grade	%	39.20
Contained Metal in Concentrate	Mlb	5,277
Contained Metal in Concentrate	kt	2,394
Key Financial Results		
Initial Capital	US$M	1,213
Mine Site Cash Cost	US$/lb Payable Cu	0.75
Total Cash Costs	US$/lb Payable Cu	1.48
Site Operating Costs	US$/t milled	53.22
After-Tax $NPV_{8\%}$	US$M	986
After-Tax IRR	%	17.2
Project Payback Period	Years	4.6

Table 22.5 Financial Results

		Before Taxation	After Taxation
Net Present Value (US$M)	Undiscounted	5,791	4,096
	4.0%	2,979	2,036
	6.0%	2,152	1,429
	8.0%	1,549	986
	10.0%	1,104	657
	12.0%	768	409
IRR	–	20.7%	17.2%
Project Payback (years)	–	4.1	4.6

Table 22.6 Mine Production and Processing Statistics

Item	Unit	Total LOM	Years 1–5	Years 6–10	LOM AVG
Ore Processed					
Quantity Ore Treated	kt	71,893	2,934	3,008	2,996
Copper Feed grade	%	3.86	4.35	4.08	3.86
Concentrate Produced					
Copper Concentrate Produced	kt (dry)	6,106	283	271	254
Copper Recovery	%	86.36	87.06	86.68	86.36
Copper Concentrate Grade	%	39.20	39.20	39.20	39.20
Contained Metal in Concentrate					
Copper	kt	2,394	111	106	100
Copper	Mlb	5,277	245	234	220
Payable Metal					
Copper	kt	2,314	107	103	96
Copper	Mlb	5,102	237	227	213

Figure 22.1 and Figure 22.2 depict the processing, concentrate and metal production, respectively.

Figure 22.1 Plant Feed Processing

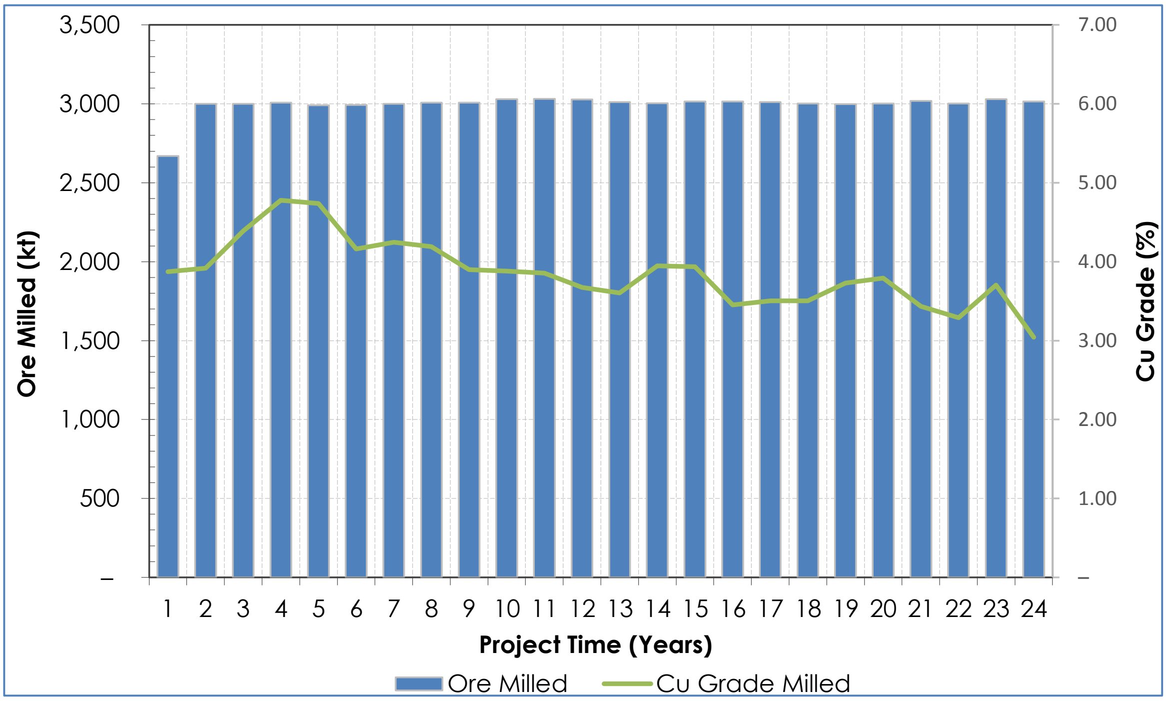


Figure by OreWin, 2016.

Figure 22.2 Concentrate and Metal Production

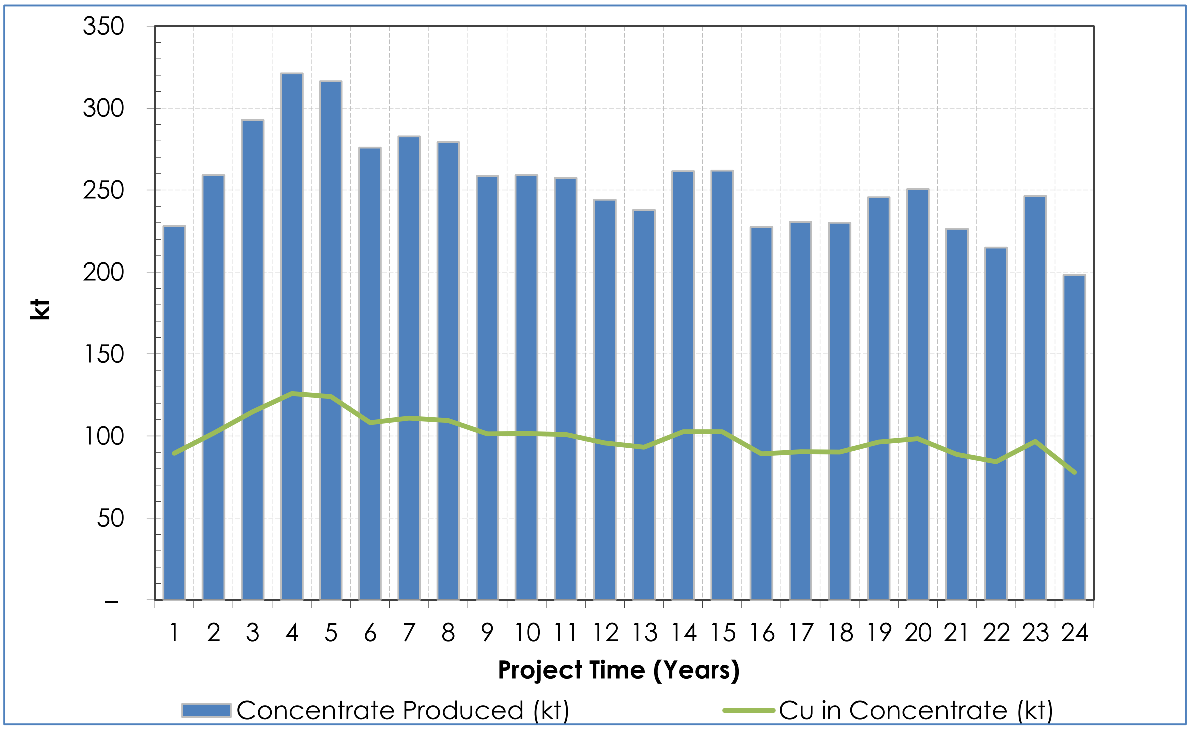


Figure by OreWin, 2016.

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

	US$/lb Contained Cu in Concentrate		
	Total Average	**Years 1–5 Average**	**Years 6–10 Average**
Mine Site	0.75	0.55	0.75
Transport	0.41	0.43	0.40
Treatment & Refining Charges	0.18	0.18	0.18
Royalties & Export Tax	0.15	0.11	0.16
Total C1 Cash Costs	**1.48**	**1.27**	**1.48**

Table 22.8 Revenue and Operating Costs

Description	Total	Years 1–5 Average	Years 6–10 Average	Total Average
	US$M	US$/t Ore Milled		
Revenue				
Copper in Concentrate	15,305	242	226	213
Gross Sales Revenue	**15,305**	**242**	**226**	**213**
Realisation Costs				
Transport	2,082	35	30	29
Treatment & Refining	911	14	13	13
Royalties & Export Tax	748	9	12	10
Total Realisation Costs	**3,741**	**58**	**56**	**52**
Net Sales Revenue	**11,565**	**184**	**170**	**161**
Site Operating Costs				
Underground Mining	2,453	24.6	37.1	34.1
Processing	886	12.4	12.3	12.3
Tailings	25	0.4	0.4	0.4
General & Administration	511	8.1	7.1	7.1
SNEL Discount	-109	-1.5	-1.5	-1.5
Customs	60	0.7	0.9	0.8
Total	**3,826**	**44.6**	**56.2**	**53.2**
Operating Margin	**7,738**	**140**	**114**	**108**
Operating Margin	**66.9%**	**75.8%**	**67.0%**	**66.9%**

The capital costs for the project are detailed in Table 22.9.

Table 22.9 Capital Investment Summary

Description	Initial Capital US$M	Sustaining US$M	Total US$M
Mining			
Underground Mining	508	499	1,007
Subtotal	**508**	**499**	**1,007**
Power			
Power Infrastructure On site	19	4	23
Power Supply Off Site	104	–	104
Subtotal	**123**	**4**	**127**
Concentrate & Tailings			
Process Plant	160	8	167
Tailings	23	62	84
Subtotal	**182**	**69**	**252**
Infrastructure			
Plant Infrastructure	18	4	23
Other Infrastructure	8	2	10
Owners Camp	10	2	12
Contractors Camp	23	5	28
Subtotal	**59**	**14**	**73**
Indirects			
EPCM	58	–	58
Subtotal	**58**	**–**	**58**
Owners Cost			
Owners Cost	95	–	95
Closure	–	67	67
Subtotal	**95**	**67**	**162**
Capital Expenditure Before Contingency	**1,025**	**655**	**1,679**
Contingency	189	44	233
Capital Expenditure After Contingency	**1,213**	**699**	**1,912**

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 NPV8 to initial capital cost, direct operating costs, transport and Cu feed grade is shown in Table 22.11. The table shows the change in the base case After Tax NPV8 of US$986 M. 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. The sensitivity to transport applies the costs via Durban (US$356/t) and via Lobito (US$300/t).

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	613	2,364	4,096	5,828	7,560
4.0%	-20	1,020	2,036	3,050	4,063
6.0%	-206	624	1,429	2,230	3,030
8.0%	-340	336	**986**	1,632	2,276
10.0%	-438	123	657	1,187	1,714
12.0%	-508	-36	409	851	1,289
15.0%	-579	-206	142	486	827
IRR	3.8%	11.5%	17.2%	22.2%	26.6%

Table 22.11 Additional Sensitivities

			Change from Base $NPV_{8\%}$ (US$M)				
Variable	**Units**	**Base Value**	**-25.0%**	**-10.0%**	**–**	**10.0%**	**25.0%**
Initial Capital	US$M	1,213	1,252	1,092	986	879	720
Direct operating costs per tonne of ore milled	US$/t	53	1,215	1,077	986	894	757
Transport costs per tonne of concentrate	US$/t	356 / 300	1,122	1,040	986	931	849
Copper feed grade	% Cu	3.86%	175	662	986	1,308	1,794

Cumulative cash flow is depicted in Figure 22.3. The Project cash flow is shown in Table 22.12.

Figure 22.3 Cumulative Cash Flow

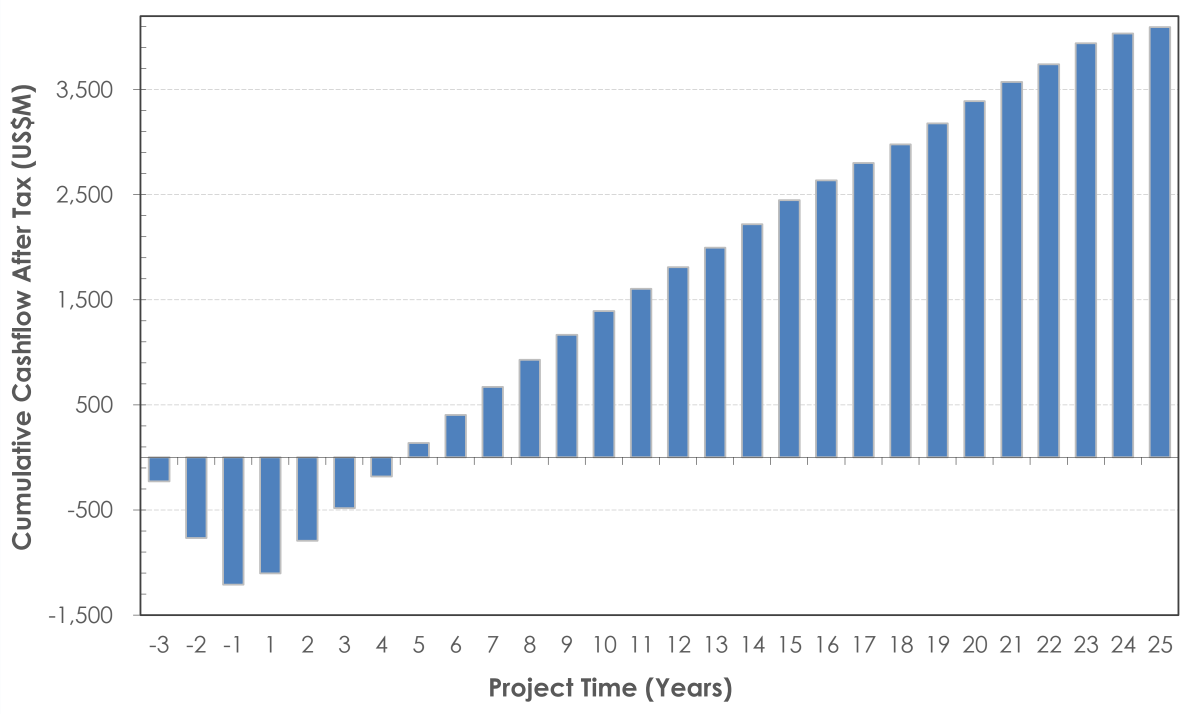


Figure by OreWin, 2016.

Table 22.12 Cash Flow

Description	Unit	Total	Year											
			-4	-3	-2	-1	1	2	3	4	5	6 to 10	11 to 20	21 to 24
Total Gross Revenue	US$M	15,305	–	–	–	–	571	649	734	805	793	3,398	6,134	2,221
Total Realisation Costs	US$M	–	–	–	–	–	–	–	–	–	–	–	–	–
Net Sales Revenue	**US$M**	**15,305**	**–**	**–**	**–**	**–**	**571**	**649**	**734**	**805**	**793**	**3,398**	**6,134**	**2,221**
Less: Site Operating Costs														
Mining	US$M	2,453	–	–	–	–	33	48	64	104	113	557	1,095	439
Processing	US$M	886	–	–	–	–	34	37	37	37	37	185	371	148
Smelting	US$M	–	–	–	–	–	–	–	–	–	–	–	–	–
Tailings	US$M	25	–	–	–	–	1	1	1	1	1	5	11	4
General & Administration	US$M	511	–	–	–	–	24	23	24	24	23	106	205	82
Discount On Power	US$M	-109	–	–	–	–	-4	-5	-5	-5	-5	-23	-46	-18
Total Site Operating Costs	**US$M**	**3,766**	**–**	**–**	**–**	**–**	**88**	**104**	**122**	**162**	**169**	**831**	**1,635**	**655**
Operating Surplus / (Deficit)	**US$M**	**11,539**	**–**	**–**	**–**	**–**	**484**	**545**	**612**	**644**	**624**	**2,566**	**4,499**	**1,566**
Less: Indirect Costs														
Depreciation	US$M	1,808	–	–	–	–	485	127	131	102	85	425	261	191
Customs	US$M	96	–	2	6	5	7	4	5	4	3	16	31	12
Total Indirects	**US$M**	**1,904**	**–**	**2**	**6**	**5**	**492**	**131**	**135**	**106**	**88**	**441**	**293**	**204**
Net Profit Before Income Tax	**US$M**	**9,635**	**–**	**-2**	**-6**	**-5**	**-9**	**414**	**476**	**538**	**535**	**2,125**	**4,206**	**1,363**
Income Tax Expense	US$M	1,694	–	–	–	–	–	–	29	103	103	392	819	249
Net Profit After Income Tax	**US$M**	**7,941**	**–**	**-2**	**-6**	**-5**	**-9**	**414**	**447**	**435**	**432**	**1,733**	**3,388**	**1,114**
Cashflow Adjustments:														
Depreciation	US$M	1,808	–	–	–	–	485	127	131	102	85	425	261	191
Capital Expenditure	US$M	-1,213	–	-226	-534	-397	-57	–	–	–	–	–	–	–
Capital Expenditure - Sustaining	US$M	-699	–	–	–	–	-143	-80	-84	-35	-9	-81	-148	-120
Working Capital	US$M	–	–	–	–	-41	-29	6	-21	-2	4	14	3	67
Net Cashflow After Tax	**US$M**	**7,837**	**-0**	**-228**	**-540**	**-443**	**247**	**467**	**472**	**500**	**513**	**2,092**	**3,505**	**1,253**
Net Cashflow Before Tax	**US$M**	**9,531**	**-0**	**-228**	**-540**	**-443**	**247**	**467**	**502**	**603**	**616**	**2,483**	**4,323**	**1,501**

22.5 Alternative Mining Method Financial Results

The basis of the mine design has assumed room and pillar mining in the shallow portions of the deposit, transitioning to drift-and-fill mining in the deeper sections.

KGHM Polska Miedź S.A. operates three large underground mines in the Legnicko-Głogowski Okręg Miedziowy (LGOM) region in Poland. The deposit at LGOM has similarities with Kamoa and employs a controlled convergence room and pillar mining method. The method allows for an increased extraction above that planned for the Base Case. The method allows for the controlled collapse of the stope backs and eliminates the requirements for fill. An initial study by KGHM Cuprum R&D Centre Ltd. (KGHM Cuprum) suggests that the controlled convergence room and pillar method would be applicable to Kamoa in both the room and pillar areas and a significant proportion of the drift-and-fill areas.

The controlled convergence room and pillar method presents an opportunity for an improvement of the financial results from the combination of increased extraction ratios, lower mining operating costs and capital costs.

As a sensitivity to the analysis in the PFS the costs for paste fill have been removed to indicate the potential financial results from adopting the controlled convergence room and pillar method. Table 22.13 shows the key financial differences between Sensitivity Case and Base Case. The results of the financial analysis for sensitivity case show an after-tax NPV8% of US$1,182 M. The case exhibits an after-tax IRR of 18.9% and a payback period of around 4.3 years. A summary of the financial results is shown in Table 22.14.

Table 22.13 Mining Method Comparison – Overall Results

Summary Results		SR&P and D&F (With Fill)	Controlled Convergence Room and Pillar (No Fill)
Item	**Unit**	**PFS Base Case**	**PFS Sensitivity**
Ore Processed			
Quantity Plant Feed Treated	kt	71,893	71,893
Copper Feed Grade	%	3.86	3.86
Concentrate Produced			
Copper Concentrate Produced	kt (dry)	6,106	6,106
Copper Concentrate Recovery	%	86.36	86.36
Copper Concentrate Grade	%	39.20	39.20
Contained Cu in Concentrate	Mlb	5,277	5,277
Key Financial Results			
Initial Capital	US$M	1,213	1,155
Mine Site Cash Cost	US$/lb Payable Cu	0.75	0.61
Realisation	US$/lb Payable Cu	0.41	0.73
Total Cash Costs After Credits	US$/lb Payable Cu	1.48	1.35
Site Operating Costs	US$/t milled	53.22	43.54
After Tax $NPV_{8\%}$	US$M	986	1,182
After Tax IRR	%	17.2%	18.9%
Project Payback Period	Years	4.6	4.3

Table 22.14 Sensitivity Case Financial Results

Comparative After Tax NPV ($M)		
Discount Rate	**SR&P and D&F (With Fill)**	**Controlled Convergence Room and Pillar (No Fill)**
	PFS Base Case	**PFS Sensitivity**
Undiscounted	4,096	4,631
4.0%	2,036	2,344
6.0%	1,429	1,672
8.0%	986	1,182
10.0%	657	819
12.0%	409	546
Internal Rate of Return	17.2%	18.9%
Project Payback Period (Years)	4.6	4.3

22.6 Cost and Production Benchmarking

The concentrate copper equivavlent production for a set of other copper projects that are currently under construction is compared with the Kamoa 2016 PFS in Figure 22.4. The capital intensity for the copper projects and the Kamoa 2016 PFS are compared in Figure 22.5. The two figures are based on data from Wood Mackenzie, February 2016. The Wood Mackenzie research identified the projects as greenfield development projects and was based on public disclosure and information gathered by Wood Mackenzie.

Figure 22.4 Copper Projects Under Construction Concentrate Equivalent Production

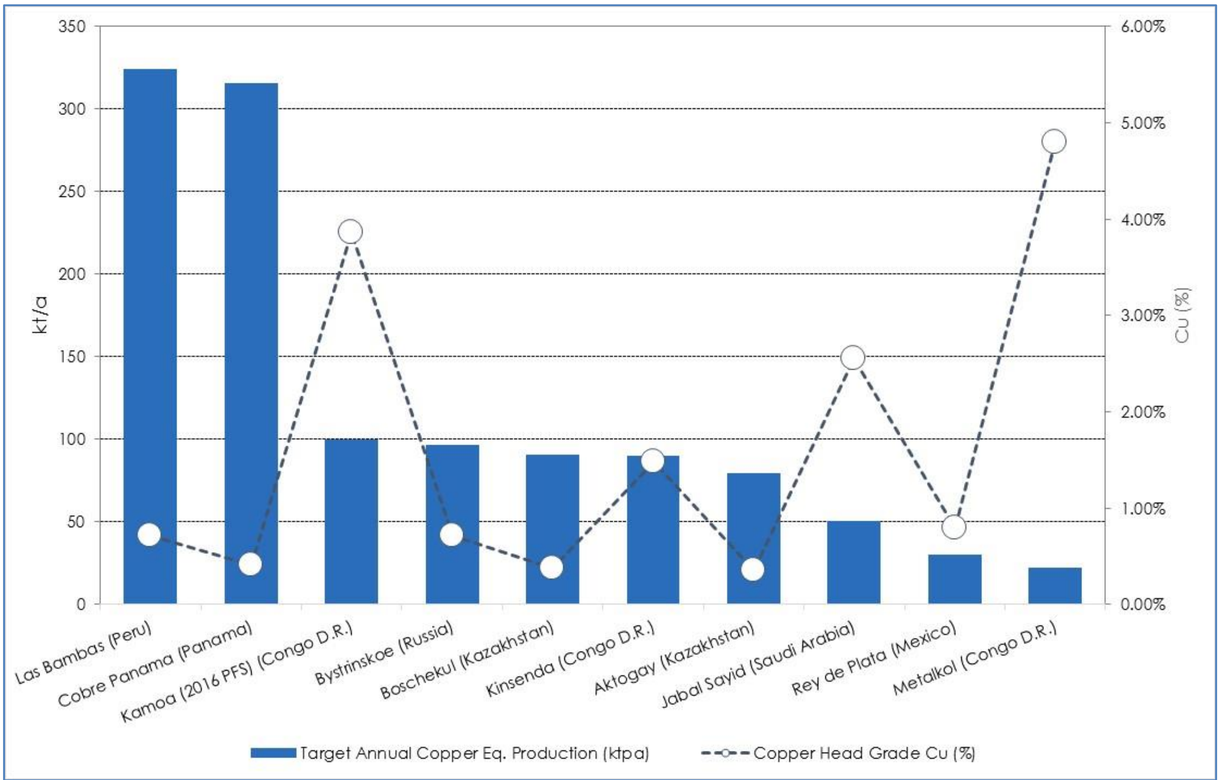


Figure based on data from Wood Mackenzie, February 2016.

Figure 22.5 Copper Projects Under Construction Capital Intensity

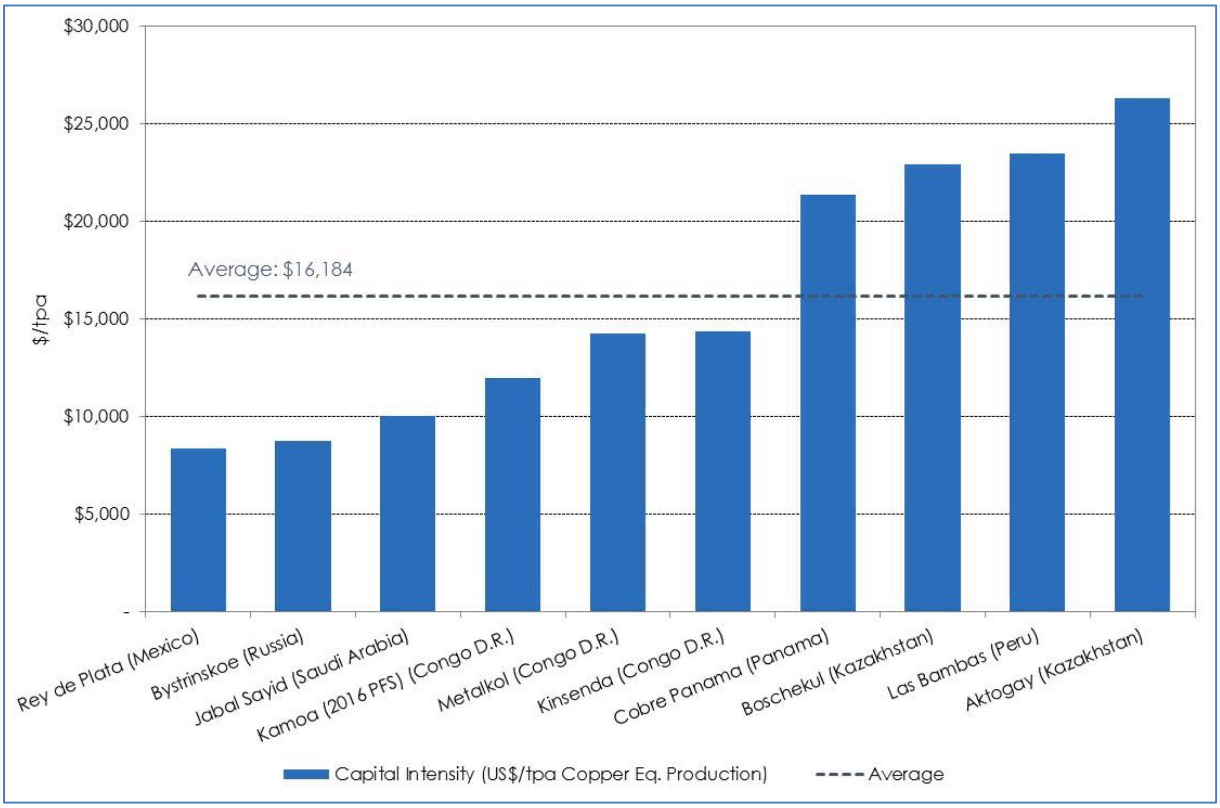


Figure based on data from Wood Mackenzie, February 2016.

23 ADJACENT PROPERTIES

There are no adjacent properties relevant to this Report.

24 OTHER RELEVANT DATA AND INFORMATION

This section not used.

25 INTERPRETATION AND CONCLUSIONS

25.1 Mineral Resource Estimate Update

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 it (Datamine macros), and has validated the resource model. Amec Foster Wheeler finds the resource model to be suitable to support prefeasibility level mine planning.

The Kakula Exploration area has excellent potential for delineation of thick, continuous, high-grade mineralisation with further drilling.

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

- Commodity prices and exchange rates.
- Cut-off grades.
- Metallurgical recoveries.
  - Metallurgical test work indicates the need for multiple grinding and flotation steps. Metallurgical variability test work has been initiated and covered has only is in early stages and has only covered a portion of the deposit. Recent test work has shown improved recoveries and concentrate grades compared to previous work.
- Mining plan.
  - The presence of 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. Mitigation plans for these risks could include an exploratory decline, or, potentially, a program of inclined drillholes.
  - Delineation drill programs 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.
  - Mining recovery could be lower and dilution increased where the dip locally increases on the flanks of the domes. An exploration decline is being driven which would provide an appropriate trial of the conceptual room-and-pillar mining method on the Kamoa deposit in terms of costs, dilution, and mining recovery. A decline would provide access to data and metallurgical samples at a bulk scale that cannot be collected at the scale of a drill sample. The extent of the decline and associated drifts are dependent on further studies.
- Infrastructure.
  - Exploitation will require building a greenfields project with attendant infrastructure.
- Capital and operating costs.
  - Development will require building a greenfields project with attendant infrastructure.

25.2 Kamoa 2016 PFS

With the additional data collection and the findings of the Kamoa 2016 PFS Ivanhoe should consider whether it is appropriate to progress studies to a feasibility stage of assessment.

25.3 Underground Mining

25.3.1 Underground Mining Methods

Mine design and associated work for the Kamoa deposit was prepared at a prefeasibility level, which requires additional engineering and design to ensure the level of design is commensurate with future financial decisions for mining the Kamoa Mineral Resource.

The Kamoa deposit is favorable for large scale mechanised room and pillar mining at shallower depths and drift-and-fill mining at depths below 500 m. Additional geotechnical data will be required to ensure the mining methods are suitable, the excavation and pillar dimensions are appropriate, and ground support requirements are adequate. Further evaluation of the potentially problematic Kamoa pyritic siltstone needs to be addressed for possible additional ground support and other remediation measures that could affect initial development productivities.

The Kamoa deposit appears to be relatively un-deformed and continuously mineralised with good local continuity and flat to moderate dips. Additional infill drilling will be required to test for the presence of minor faults, which may result in changes in the roof and floor positions, increased dilution, and increased waste development. This drilling will also assist in validating local dip changes that may result in lower productivities. This drilling, along with further detailed mineralisation continuity studies, should help validate hanging and footwall continuity.

Additional hydrological data and studies will be required to confirm that groundwater inflows will not significantly impact underground costs and schedules. The water handling allowance included in the Kamoa 2016 PFS and the effects on ground conditions and productivities can then be further assessed.

25.3.2 Underground Mining Accesses

The thickness of the Kalahari sand in the overburden sequence may dictate the location of the ventilation raises and increase the costs of ventilation for the mine.

25.3.3 Production Schedules

Although the deposit appears to provide significant resources, and mining will be conducted in multiple sections, there may be a risk of having to schedule additional mining areas to ensure that the number of required active stopes is met and maintained. This could increase the ramp up periods for build up to full production.

Backfill requirements of approximately 4.0 Mtpa produces logistical complication that must be further analysed in determining the drift-and-fill mining rates. Further design of the backfill, along with the availability of material to be used in the paste backfill, must also be analysed.

Total personnel requirements for the mine operation are estimated to be in excess of 1,000 persons. Sourcing and training a skilled workforce to achieve this level presents challenges, especially in a remote location.

25.3.4 Logistics

Logistics to and from the DRC will present a significant challenge to the project during construction and operation. Significant effort is required during subsequent stages of the project to adequately prepare for these challenges.

25.3.5 Risk

The risks and uncertainties described below are not the only risks and uncertainties that Kamoa 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, Kamoa could be materially and adversely affected.

- Kamoa may not achieve its production estimates, and the development into a commercially viable mine cannot be assured.
- Kamoa 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 determinants that are needed to develop a mine. The failure to address these infrastructure requirements could affect the ability to commence or continue production.
- In addition, unusual or infrequent weather phenomena, 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.
- Kamoa will require approvals, licences and permits that it currently does not have, in order to continue its exploration and development activities, and if deemed viable, commence mining operations.
- Kamoa 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 development and success of the Project 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 the Company 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.
- 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 the Company'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 the Company.
- 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

With the additional data collection and the findings of the Kamoa 2016 PFS Ivanhoe is in a position to progress studies to a feasibility stage of assessment. Further study work should be undertaken to optimise the project production rate by considering concentrator and smelter capacities that are matched to the power supply availability, mine production and transport options. Based on the results of the Kamoa 2016 PFS, key changes that should be considered in the Feasibility Study are:

- Change the mining method to controlled convergence room and pillar stoping.
- Evaluate increased production up to 4 Mtpa from the proposed initial mining area using the controlled convergence room and pillar mining method and limited adjustments to the ore handling and ventilation systems.
- Continue to evaluate the options for rail transport to Lobito.

The following is a list of key activities for further work:

- Continue infill drilling programme to upgrade resource categorization, 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 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.
- 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.

26.2 Drilling

Amec Foster Wheeler has recommended a drill program that totals approximately 40,500 m, at a cost of about US$4.2 M. The program will have two objectives.

The first objective is exploration and infill drilling. Drillholes are proposed at Kakula for both hypogene and shallow supergene targets. Shallow drilling is recommended to be conducted at Kamoa Nord, Makalu NW, Kakula, Kansoko Nord, and around domes. In addition, deeper exploration drillholes are suggested to test other targets.

The second objective is to obtain additional information to support detailed studies. Additional engineering drillholes, including metallurgical drilling should be completed within the defined Kamoa Mineral Resource area.

26.3 Programme Details

Figure 26.1 is a proposed drill location plan for the Project. Green triangles represent shallow Land Cruiser (LC) supported drilling, red squares represent deep drilling and black crosses represent completed drillholes. The drill collar locations are provisional, and it is expected that Ivanhoe will continue to refine both the selected drill target areas, and the collar locations as results of each drillhole become available.

A drilling cost of US$102.70/m was used for the proposed drilling budget outlined in Table 26.1. The overall programme total of US$4.2 M includes provision for the purchase of a dedicated drilling rig for US$930,000.

Figure 26.1 Kamoa Provisional Drill Programme

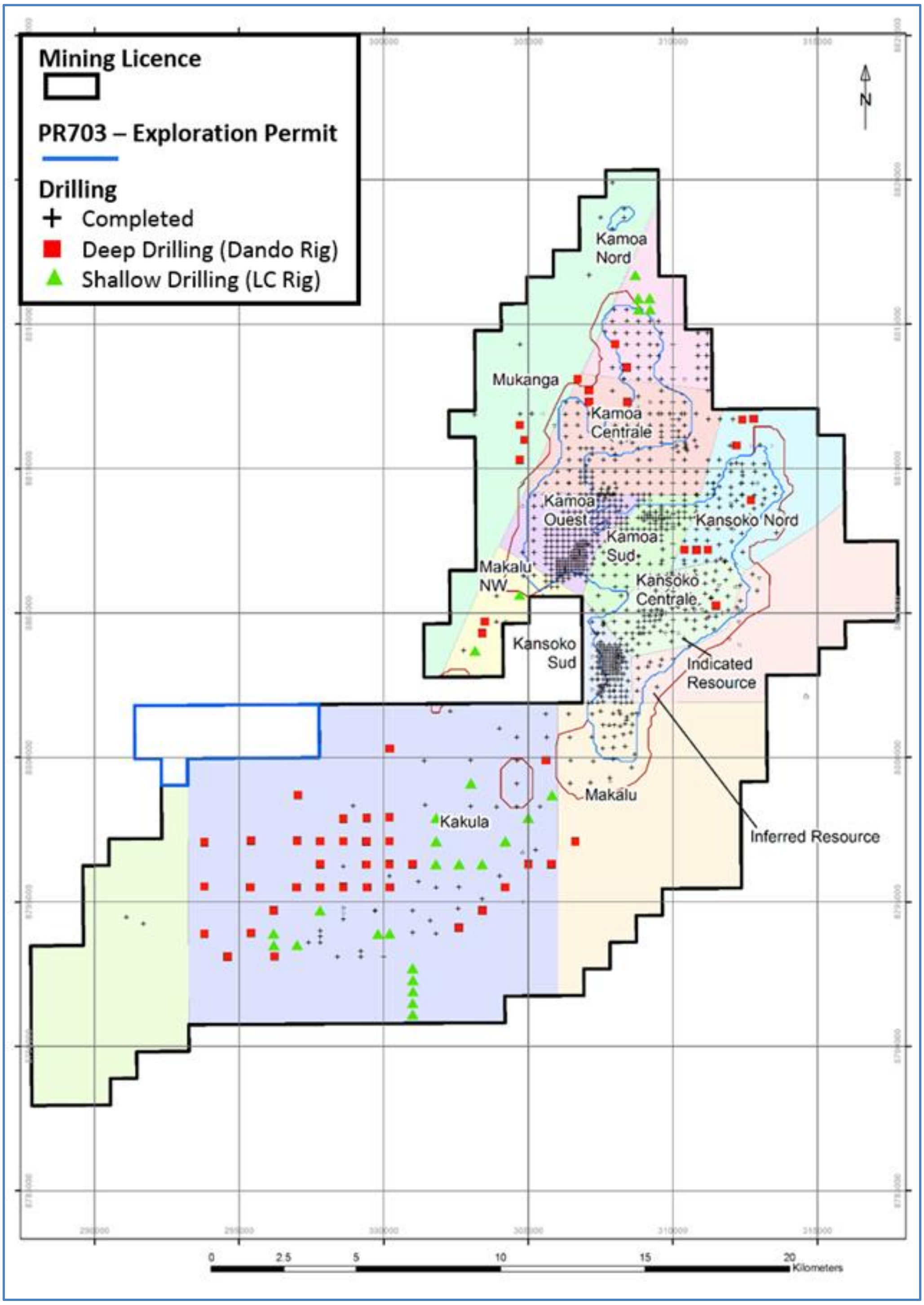


Figure by Ivanhoe, modified by Amec Foster Wheeler, 2016. See Figure 14.23 for explanation of exploration areas colored above.

Table 26.1 Kamoa Provisional Drill Programme

Program	Proposed Drillholes	Proposed Metres
Land Cruiser mounted rig (includes exploration and brownfields)	50	4,175
Exploration	47	20,030
Infill, geotechnical and metallurgical drilling	37	16,300
Subtotal	134	40,505

26.4 Underground Mining

The following is a list of mining recommendations for the Kamoa project.

- Investigate the suitability of Controlled Convergence Room and Pillar as part of the feasibility study.
- Continue infill drilling programme to upgrade resource categorization, 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 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.
- Undertake backfill plant design along with an evaluation and sourcing of suitable materials for paste fill.
- 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.

26.5 Process Plant

The following is a list of process recommendations for the Kamoa project.

- Kamoa Copper SA develop a reliable and economic measurement method to determine the copper mineralogy of samples. Without such a tool, and without implementing such a tool in the block model, it will not be possible to reliably predict concentrate grades and (to a lesser extent) copper recoveries.
  - Ivanhoe are internally evaluating the use of existing geological Cu:S ratio data for delineating domains within the mineralogical spectrum from pyrite through chalcopyrite, bornite, chalcocite and oxide/native copper. Verification of the predictive power of the Cu:S ratio by flotation testing is expected to cost in the order of $85 000.
- 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 controlling silica levels in the final concentrate regardless of

copper sulphide mineralogy.

- This work may be partially or fully combined with the verification work proposed in the previous recommendation. If conducted separately this program is expected to cost in the order of $60 000

- 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 test program in which a further 20 samples (each consisting of 20 particles) were CWI tested would be expected to cost in the order of $10 000.

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