Ivanplats Limited

Kamoa Copper Project

Katanga Province, Democratic Republic of Congo

NI 43-101 Technical Report on Updated Mineral Resource Estimate

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Effective Date: 11 March 2013

Project Number: 172449

CERTIFICATE OF QUALIFIED PERSON

I, Harry Parker, Ph.D., RM SME., am employed as a Technical Director with AMEC E&C Services, Inc.

This certificate applies to the technical report entitled "Kamoa Copper Project, Katanga Province, Democratic Republic of Congo, NI 43-101 Technical Report on Updated Mineral Resource Estimate", that has an effective date of 11 March 2013 (the "Technical Report").

I am a Professional Geologist in California (#3402), in Arizona (#13317), and in Minnesota (#49606), and a Fellow of the Australian Institute of Mining and Metallurgy. I am also a Registered Member of the Society for Mining, Metallurgy and Exploration (#2460450). I graduated from Stanford University with BSc and PhD degrees in Geology in 1967 and 1975 respectively. I graduated from Harvard University in 1969 with an AM degree in Geology. I graduated from Stanford University with an MSc degree in Statistics in 1974.

I have practiced my profession for 46 years during which time I have been involved in the estimation of mineral resources and mineral reserves for various mineral exploration projects and operating mines. I have previously estimated or audited Mineral Resources for sediment-hosted copper deposits, including Konkola, Nchanga, Chambishi, and Mufilira in Zambia, and have prepared Competent Person's reports on the entire mining operations of the Zambian Copperbelt.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I visited the Kamoa Project (the "Project") between 1 and 3 May, 2009 from 27 to 30 April 2010, and again from 12–14 November, 2012.

I am responsible for Sections 1.1.1, 1.2 to 1.8, 1.10 to 1.13, 1.15.1, 1.16, 2, 3, 4, 5, 6, 7, 8, 9.1 to 9.5, 9.8, 9.9, 10.1 to 10.6, 10.10 to 10.12, 11 (excepting 11.3), 12.1 to 12.4, 12.6, 14, 15, 20, 23, 24, 25.1, 26 and 27 of the Technical Report:

I am independent of Ivanplats Limited as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Kamoa Project since 2009 during which time I have prepared mineral resource estimates on the Project.

I have previously prepared a technical report on the Project as follows:

Peters, B.F, Parker, H.M., Seibel, G., Jakubek, J., Annavarapu, S., and David, D., 2012: Ivanplats Limited, Kamoa Project, Katanga Province, D.R.C., NI 43-101 Technical Report: unpublished technical report prepared for Ivanplats Limited by AMC Consultants, AMEC E&C Services Inc., SRK Consulting Inc and Stantec Inc., effective date 5 September 2012, 375 p.

I have read NI 43–101 and those portions of the Technical Report for which I am responsible have been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: 28 March, 2013

“signed and sealed”

Dr Harry Parker, RM SME.

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I, Gordon Seibel, RM SME, am employed as a Principal Geologist with AMEC E&C Services Inc.

This certificate applies to the technical report entitled "Kamoa Copper Project, Katanga Province, Democratic Republic of Congo, NI 43-101 Technical Report on Updated Mineral Resource Estimate", that has an effective date of 11 March 2013 (the "Technical Report").

I am a Registered Member of the Society for Mining, Metallurgy and Exploration (#2894840). I graduated from the University of Colorado with a Bachelor of Arts degree in Geology in 1980. In addition, I obtained a Masters of Science degree in Geology from Colorado State University in 1991.

I have practiced my profession for 33 years, during which time I have been directly involved in the development of resource models and mineral resource estimation for mineral projects in North America, South America, Africa, and Australia since 1991. I have previously estimated or audited Mineral Resources for sediment-hosted copper deposits, including Konkola North, Lupoto, and Kakanda.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I visited the Kamoa Project (the "Project") from 9 to 10 February 2011, from 5 to 8 November 2011, and again from 12–14 November, 2012.

I am co-responsible for Sections: 1.10, 1.11, 1.12, 1.13, 1.15.1, 2, 3, and 14 of the Technical Report.

I am independent of Ivanplats Limited as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Kamoa Project since late 2010 during preparation of mineral resource estimations.

I have previously prepared a technical report on the Project as follows:

Peters, B.F, Parker, H.M., Seibel, G., Jakubek, J., Annavarapu, S., and David, D., 2012: Ivanplats Limited, Kamoa Project, Katanga Province, D.R.C., NI 43-101 Technical Report: unpublished technical report prepared for Ivanplats Limited by AMC Consultants, AMEC E&C Services Inc., SRK Consulting Inc and Stantec Inc., effective date 5 September 2012, 375 p.

I have read NI 43–101 and those portions of the Technical Report for which I am responsible have been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, those portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: 28 March, 2013

“signed and sealed”

Gordon Seibel, RM SME

CERTIFICATE OF QUALIFIED PERSON

I, Dean David, FAusIMM, am employed as Technical Director, Process with AMEC Australia Pty Ltd (AMEC).

This certificate applies to the technical report entitled "Kamoa Copper Project, Katanga Province, Democratic Republic of Congo, NI 43-101 Technical Report on Updated Mineral Resource Estimate", that has an effective date of 11 March 2013 (the "Technical Report").

I am a Fellow of The Australasian Institute of Mining and Metallurgy (membership number 102351). I graduated from The South Australian Institute of Technology (now University of South Australia) with a Bachelor of Applied Science in Metallurgy in 1982.

I have practiced my profession for 31 years. I have been directly involved in mineral processing research, operations, management and consulting, specializing in metallurgical testwork program design and review, comminution, classification, flotation, geometallurgy, beneficiation, dense media separation, and mine-mill optimization for copper projects in Australia, Asia-Pacific, Africa, and South America.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I visited the Kamoa Project ("the Project") between the following dates: April 27 and April 30 2010, and 13 to 15 April 2011.

I am responsible for Sections 1.9, 2, 3, 9.7, 10.9, 11.3, 13, 17, 18, 21.5, and 21.6 of the Technical Report.

I am independent of Ivanplats Limited as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Project since March 2010, during which time I have been involved with review of metallurgical testwork and co-preparation of a Preliminary Economic Assessment study.

I have previously prepared a technical report on the Project as follows:

Peters, B.F, Parker, H.M., Seibel, G., Jakubek, J., Annavarapu, S., and David, D., 2012: Ivanplats Limited, Kamoa Project, Katanga Province, D.R.C., NI 43-101 Technical Report: unpublished technical report prepared for Ivanplats Limited by AMC Consultants, AMEC E&C Services Inc., SRK Consulting Inc and Stantec Inc., effective date 5 September 2012, 375 p.

I have read NI 43–101 and those portions of the Technical Report for which I am responsible have been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: 28 March, 2013

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CERTIFICATE of AUTHOR

This Certificate of Author has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects as published 30 June 2011, Part 8.1.

a) Name, Address, Occupation:
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AMC Consultants Pty Ltd, 4 Greenhill Road, Wayville South Australia, 5034, Australia
Principal Mining Engineer, employed as Mining Manager.

b) Title and Date of Technical Report:
Kamoa Copper Project, Katanga Province, Democratic Republic of Congo, NI 43-101 Technical Report on Updated Mineral Resource Estimate dated 11 March 2013 (the "Technical Report")

c) Qualifications:
I graduated from the University of Melbourne, Australia with a Bachelor of Engineering in Mining Engineering in 1986. I am a Fellow of the Australasian Institute of Mining and Metallurgy (no. 201743). I have practised my profession continuously since 1986 and have experience in mining operations and consulting at and for mines in various countries including Australia, Bolivia, Indonesia, Kazakstan, Kyrgyzstan, Mongolia, Peru, Philippines and Russia. As a result of my qualifications and experience, I am a Qualified Person as defined in National Instrument 43-101.

d) Site Inspection:
Bernard Peters visited the property in visited the site from 15 February 2010 to 17 February 2010, from 27 to 30 April 2010 and on 15 November 2012. The site visits included briefings from Ivanplats 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.

e) Responsibilities:
I was responsible for: Sections 1.1.2,1.14 and1.15.2; Section 2; Section 15; Sections 16.1, 16.3 and 16.5; Section 19; Sections 21.1, 21.2, 21.3, 21.7 and 21.8; Section 22; Section 25.2.

f) Independence:
I am independent of Ivanplats Limited in accordance with the application of Section 1.5 of National Instrument 43-101.

g) Prior Involvement
I have been working on various aspects of the studies of the project since 2008.

h) Compliance with NI 43-101
I have read National Instrument 43-101 and Form 43-101FI and the Technical Report has been prepared in compliance with same.

i) Disclosure
As of 28 March 2013, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated, .

/s/Bernard Peters

Bernard Peters

CERTIFICATE OF QUALIFIED PERSON

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I, Jarek Jakubec, C. Eng. (UK), MIMMM, am employed as a Corporate Consultant (Rock Mechanics), with SRK Consulting.

This certificate applies to the technical report entitled "Kamoa Copper Project, Katanga Province, Democratic Republic of Congo, NI 43-101 Technical Report on Updated Mineral Resource Estimate", that has an effective date of March 11, 2013(the "**Technical Report**").

I am a registered Chartered Engineer (No 509147) and member of the Institute of Materials, Minerals and Mining in the United Kingdom. I am a graduate of the Mining University in Ostrava, Czech Republic with a MSc. in Mining Geology (1984).

I have practiced my profession continuously since 1984 and I have 25 years' experience in mining. I have been involved in project management, mine design, due diligence studies, geological and geotechnical modeling around the world. I have direct operational experience from caving mines in Canada and have been involved in caving or sublevel caving mines studies in Canada, the United States, Chile, South Africa, Australia, Indonesia, Papua New Guinea, China and Mongolia.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects ("**NI 43–101**").

I visited the Kamoa Project ("the **Project**") between April 27, 2010 and April 30, 2012.

I am responsible for Section 9.6, 10.6.1, 10.7, 10.8, 12.5, and 16.2 of the Technical Report.

I am independent of Ivanplats Limited as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Project since 2007, and have performed underground mining studies and underground cost estimates.

I have read NI 43–101 and those portions of the Technical Report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

"Original signed"
Jarek Jakubec, C. Eng. (UK), MIMMM
Dated:

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CERTIFICATE of AUTHOR

This Certificate of Author has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects as published 30 June 2011, Part 8.1.

a) Name, Address, Occupation:
Mel Lawson
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Principal / Senior Consulting Engineer, employed as Mining Principal.

b) Title and Date of Technical Report:
Kamoa Copper Project, Katanga Province, Democratic Republic of Congo, NI 43-101 Technical Report on Updated Mineral Resource Estimate dated 11 March 2013 (the "Technical Report").

c) Qualifications:
I graduated with the degree of Bachelor of Science in Mining Engineering from the Michigan Technological University in Houghton, Michigan, in 1970. I have practiced my profession since 1970. I have an extensive background in domestic and international mine management, with an emphasis on underground and surface mining operations, including 23 years in senior operations and project management roles in various countries including Australia, Canada, Chile, Democratic Republic of Congo, New Zealand, South Africa, United States, and Uzbekistan. I am a Registered Member (1859650) of the Society for Mining, Metallurgy, and Exploration, Inc. As a result of my qualifications and experience, I am a Qualified Person as defined in National Instrument 43-101.

d) Site Inspection:
Mel K. Lawson visited the site on 15 November 2012. During the site visit, Mr. Lawson attended a briefing by local staff, undertook a camp tour, inspected the drill core (both general and from the proposed Centrale portal), and conducted a general site reconnaissance driving tour and inspection of the proposed Centrale portal. Mr. Lawson also attended a Kamoa Project kick-off meeting in Johannesburg from 12-13 November 2012, prior to visiting site.

e) Responsibilities:
I am responsible for: Section 2, Section 16.4, and Section 21.4.

f) Independence:
I am independent of Ivanplats Limited in accordance with the application of Section 1.5 of National Instrument 43-101.

g) Prior Involvement
I have been working on various aspects of the studies of the project since 2012.

h) Compliance with NI 43-101
I have read National Instrument 43-101 and Form 43-101FI and the Technical Report has been prepared in compliance with same.

i) Disclosure
As of 28 March 2013, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 28th day of March 2013.

/s/ Mel K. Lawson
Signature
Mel K. Lawson
Stantec Consulting International LLC

IMPORTANT NOTICE

This report was prepared as National Instrument 43-101 Technical Report for Ivanplats Limited (Ivanplats) by AMEC E&C Services Inc. (AMEC). The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in AMEC's services, 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. This report is intended for use by Ivanplats subject to terms and conditions of its contract with AMEC. Except for the purposed legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party is at that party's sole risk.

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1.0 SUMMARY

AMEC E&C Services Inc. (AMEC) has prepared an independent Technical Report (the Report) on the Kamoa Copper Project (the Project) located in the Democratic Republic of Congo (DRC) for Ivanplats Limited (Ivanplats).

The Report was prepared in support of Ivanplats' Annual Information Form for the year ended 31 December, 2012.

The Report documents a Mineral Resource estimate update, and also includes a summary of the Preliminary Economic Assessment study completed on the Project with an effective date of 5 September 2012 (the 2012 PEA). Results from the 2012 PEA mining studies have not changed in terms of their outcomes as their underlying assumptions remain reasonable. By inspection, the tonnage and grade of the material included in the 2012 PEA mine plan that is based on the September 2011 Mineral Resource estimate is not materially different to the tonnage and grade estimated for the same area in the December 2012 resource update.

In addition to AMEC, the following companies were involved in the preparation of the 2012 PEA:

- AMC Consultants: Overall report preparation, financial model
- Stantec Inc.: Underground mine plan
- Golder Associates: Hydrogeology and water
- African Mining Consultants: Environmental
- SRK Consulting Inc.: Underground geotechnical recommendations.

1.1 Key Findings

1.1.1 Mineral Resource Update

- 739 Mt of Indicated Mineral Resources at 2.67% Cu and 227 Mt of Inferred Mineral Resources at 1.96% Cu estimated using a 1% Cu cut-off and minimum 3 m vertical thickness. These Mineral Resources could be mined using underground or open-pit methods.
- 96 Mt of Indicated Mineral Resources at 0.69% Cu and 10 Mt of Inferred Mineral Resources at 0.69 % Cu that have been estimated using a 0.5% Cu cut-off and that occur adjacent to Mineral Resources meeting a 1% Cu cut-off over a minimum

3 m vertical thickness and have a maximum depth extent of 100 m. These Mineral Resources could be mined using open-pit methods.

- An Exploration Target that could contain from 520 Mt to 790 Mt grading 1.6% Cu to 2.5% Cu. AMEC cautions that the potential quantity and grade of this 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. Tonnages and grades estimates were estimated by tabulating the Domain 2 model and applying a ±20% variance to the tonnages and grades.

1.1.2 2012 PEA

As the 2012 PEA is still considered to be current, the following work and outcomes from the 2012 PEA are considered to remain reasonable as their underlying assumptions have not changed.

- The mining rate and concentrator feed capacity is 5 Mtpa, producing 143 ktpa of payable copper on average in the first 10 years of operation.
- The production scenario schedules 299 Mt over 61 years, producing 7.8 Mt of blister copper
- The long term price assumption is US$2.85/lb for copper
- The economic analysis returns a Net Present Value (NPV) at a 10% discount rate of US$1.16 billion (after tax).
- It has an after tax internal rate of return (IRR) of 17.0% and a payback period of 5.29 years.
- The average total cash cost after credits in the first 10 years of production is US$0.95/lb of copper.

The above-listed key outcomes contain forward-looking information. The assumptions and risks in relation to the forward-looking information are summarized and explained in more detail in Section 1.14 and Section 22 of the Report. The 2012 PEA includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorized as Mineral Reserves, and there is no certainty that the results will be realized.

1.2 Property Description and Location

The Kamoa Project is situated in the Kolwezi District of Katanga 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.

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 Ivanplats 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 12 km from the central part of the area where Mineral Resources have been estimated.

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, Ivanplats successfully conducted exploration programs 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. Ivanplats 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.

1.3 Mineral and Surface Rights, Royalties, and Agreements

The Project consists of three granted Exploitation Permits 12873, 13025 and 13026 and one Exploration Permit, 705. The three Exploitation Permits cover an area of 397.4 km^2, and the Exploration Permit covers 1.7 km^2 for a total Project area of 399.1 km^2.

The Project is 95% held in the name of African Minerals (Barbados) Ltd. sprl (AMBL), a wholly-owned Ivanplats subsidiary. On 11 September, 2012, Ivanplats transferred a 5% non-dilutable interest in the share capital of AMBL to a DRC state-owned nominee. Ivanplats 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, Ivanplats 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 Ivanplats.

Land access for the exploration programs completed to date has typically been negotiated without problems. Where compensation has been required for exploration activities, compensation has followed 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 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. Different rates apply to different types of metals sold.

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.

The DRC Government is currently considering an update to the 2002 Mining Code. Legislation is currently in draft form.

1.4 Geology and Mineralization

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

The regional geology comprises sediments 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 mineralization form a sequence known as the Katanga Supergroup, comprising the Roan, Lower Kundelungu, and Upper Kundelungu Groups. Copper mineralization 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 mineralized sequence at the base of the diamictite comprises several interbedded units which appear to control copper mineralization. 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 mineralization upward, the copper sulphides in the mineralized 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) mineralization. Most of the higher-grade mineralization 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. Mineralization comprises three distinct styles: supergene, mixed, and hypogene mineralization.

Near the surface adjacent to the Kamoa Dome, the diamictites have been leached, resulting in local 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 mineralization, which has been encountered to depths of 400 m in drill holes collared near faults, analogue deposits in Zambia such as Mufulira East, Nkana, and Chibuluma South had relatively narrow zones of oxides/supergene enrichment near surface. Hypogene mineralization occurs at depths as shallow as 30 m.

1.5 History and Exploration

Although exploration was undertaken by the Tenke Fungurume Consortium in the period 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 Ivanplats 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 preliminary economic assessment. An updated preliminary economic assessment is in progress.

In the opinion of the AMEC QPs, the exploration programs completed to date are appropriate to the style of the Kamoa deposit. The provisional research work that has been undertaken supports Ivanplats' deposit genetic and affinity interpretations for the Project area. The Project area remains prospective for additional discoveries of base-metal mineralization around known dome complexes. Anomalies generated by geochemical, geophysical and drill programs to date support additional work on the Project area.

1.6 Drilling

The drill database used for Mineral Resource estimation was closed at 10 December 2012. Drilling has been ongoing since that date, and information on the program to 11

March 2013 is included in this Report. Drill hole collar locations are shown for the entire Project in Figure 10-1. Drilling is summarized in Table 10-1. For the remainder of this report, the 15 RC pre-collar (RD) holes are included with the core drill holes in any reference to core drilling. Two core holes were being drilled as at 11 March 2013.

As at 11 March 2013, there were 842 core holes within the Project area boundaries (including five close-spaced wedged deflections and six twin holes), Figure 10-2. Of these drill holes, a total of 555 are used to support the December 2012 Mineral Resource estimate (543 drill holes are in Domain1 (area where Indicated and Inferred Mineral Resources are estimated) plus 12 are in Domain 2 (the area where the Exploration Target is estimated).

At the time of the completion of the September 2011 resource estimate on which the 2012 PEA is based, a total of 720 holes lay within the boundaries of the then-defined Project; 309 core holes (including five close-spaced wedged deflections and six twin holes) are spaced close enough together (spacing less than approximately 800 m) to allow Mineral Resource estimation (Figure 10-3). A total of 330 core holes were used to model the stratigraphy within the Kamoa Project for the September 2011 resource estimate.

Standard 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 mineralized units ranges from 0% to 100% and averages 95%. The 0% recovery is likely due to missing data, as logging does not indicate poor recovery. Visual inspection by AMEC documented the core recovery to be excellent. From 2010 through 2012, all completed holes were surveyed by an independent professional surveyor, SD Geomatique, using a differential GPS that was accurate to within 20 mm. Down-hole surveys for most drill holes were performed by the drilling contractor at approximately 30 m intervals for 2009 drilling and at 50 m intervals for 2010 through 2013 drill holes, using a Single Shot digital down-hole instrument.

The quantity and quality of the lithological, geotechnical, collar, and down-hole survey data collected in the core drill programs is sufficient, in the opinion of the AMEC QPs, to support Mineral Resource estimation.

1.7 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 mineralization and/or the mineralized 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 mineralized basal diamictite 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. A 3 m shoulder is left above the first visible sign of copper mineralization in each drill hole.

A total of 10,111 specific gravity measurements were performed on samples taken from drill core using a water immersion method.

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. ALS Vancouver, British Columbia, acted as the check laboratory for drill core samples from part of the 2009 program and for 2010 through 2012 drilling. ALS Chemex is ISO:9000:2008 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 pulverized in the same manner as the RC samples. 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, 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 analyzed 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 are now being undertaken.

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

1.8 Data Verification

AMEC reviewed the sample chain of custody, quality assurance and control procedures, and qualifications of analytical laboratories. AMEC is of the opinion that the procedures and QA/QC control are acceptable to support Mineral Resource estimation. AMEC also audited the assay database, core logging, and geological interpretations.

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

1.9 Metallurgical Testwork

The first metallurgical testwork on Kamoa mineralization was carried out in 2010 using drill cores from the heavily explored, and shallow, Kamoa Sud area of the deposit. The samples were subjected to comminution and flotation tests at Mintek laboratories in Johannesburg.

Comminution tests indicated that the mineralized rock is competent with respect to SAG milling, that the Bond ball mill work index results are modest and in the range of 14 to 16 kWh/t and that the mineralization is moderately abrasive. Due to milling efficiency considerations, mineralized materials with these properties, and at treatment rates of less than about 6 Mt/a, are best processed in crush/ball mill circuits rather than SAG circuits. These comminution results show that the Kamoa mineralization is moderately harder than typical Copperbelt ores.

The Mintek flotation testwork showed that the mineralization was amenable to treatment by conventional sulphide flotation but with the proviso that a significant amount of regrinding is required. Flotation recoveries were lower than expected for Copperbelt ores due to a non-floating copper sulphide population locked in silicates at sulphide phase sizes of 10 µm or finer. The economic copper minerals include chalcopyrite, bornite and chalcocite.

Although this phase of the testwork was preliminary, it did indicate that an effective flotation flow sheet will include roughing, cleaning and multiple stages of re-cleaning after fine regrinding.

An acceptable smelter-grade re-cleaner concentrate containing 27% Cu, 15% Fe, 17% S and 26% SiO_2 was produced at a copper recovery of 79% on a preliminary master composite sample obtained from the mixed supergene and hypogene zones.

Subsequent testwork in 2011 at Xstrata Process Support (XPS) laboratories in Sudbury, Canada, on new and more representative samples, has shown that the ultimate Mintek flow sheet was an appropriate starting point for further optimisation work. Consequently higher concentrate grades have been achieved at improved recoveries. The XPS work forms the basis for the 2012 PEA flow sheet concept.

The best XPS results on hypogene mineralization were achieved in a circuit utilising two regrind stages, one to 15 µm and the second to 10 µm. The circuit achieved a copper recovery of 85.4% at a grade of 32.8% Cu.

New composites of hypogene and supergene mineralization were prepared by XPS during 2012. Testing of these samples included changing from ball mill grinding to rod mill grinding; changing the regrind size distribution P80 targets; conducting primary and secondary rougher flotation under alkaline conditions rather than natural acidic conditions and conducting locked cycle testing with a modified circuit. Copper recovery improvements of between 2% and 5% have been noted, partially as a result of using rod milling and partially as a result of using alkaline flotation conditions. However, additional testwork is required before it is possible to modify the existing recovery values and equations that are documented in the 2012 PEA.

The tested Kamoa mineralized materials display a very high degree of competence and high grindability indices. The choices for milling mineralization of this type include single-stage SAG mills, Secondary crush SAG milling, semi autogenous-ball mill-crushing (SABC) circuits, HPGR circuits and conventional cone crush and ball mill circuits. At the throughput rates envisaged in the 2012 PEA, a multiple stage crush circuit followed by ball milling was recommended. Consideration could be given to the third crushing stage being an HPGR, but given the project location the simplicity of cone crushing is likely to provide a more robust solution. It is expected that the comminution circuit will be re-evaluated in the updated PEA that is ongoing.

AMEC notes that work at XPS has been initiated on supergene mineralization, but not finalised. When supergene work is complete, a set of updated algorithms will be developed, which are likely to vary from those used in the 2012 PEA.

A program of metallurgical variability testing has not yet been conducted but samples have been identified for this work. Testwork to date has shown no penalty elements present to problematic levels in the concentrate. The main impurity element is silica, and testwork is being undertaken to minimise silica recovery. Saleable concentrate generated from the XPS test program in February 2012 averaged 32.5% Cu and contained 80 ppm As, 1.1% MgO, 5.0% Al_2O_3 and 19.0% SiO_2.

It can be assumed there will be no impact on the proposed plant performance assumptions, or the capital and operating cost estimates in the 2012 PEA. The additional testwork results have provided the following specific assurances and cautions:

- The sample competence is higher than was previously assumed, but this only strengthens the case for preferring a crush and ball mill circuit over a SAG mill based circuit.
- Higher competence may result in additional crushing and screening requirements but crushing design is typically based on Bond Crusher Work index results (CWI), none of which are currently available. Extraction of suitable diameter core for CWI measurement is recommended before a definitive crusher design is possible.
- The design grindability values (BWI and RWI) have not changed significantly with the new results. The most serious implication from the new results is the negative effect that the harder pyritic siltstone values will have on mill throughput. When drawing from the few mining locations where pyritic siltstone is the main mineralised material, then its proportion in mill feed will need to be monitored and controlled.
- The Bond abrasion index used in design is conservative
- The copper recovery from hypogene material is expected to improve above the current results once new test conditions are established
- The copper recovery and concentrate grade from supergene mineralized samples continues to be variable, and a program of variability testing is needed to define the nature of supergene mineralization across the resource.

1.10 Mineral Resource Estimates

The cut-off date for exporting the drill holes from the database was 10 December 2012. The perimeter of the mineralization was defined using 555 mostly vertical, mineralized core drill holes that excluded barren leached drill holes where the mineralization approaches the surface.

The best single mineralized intercept (SMZ) for each of the 555 holes within the resource boundary was selected using the criteria of a minimum copper grade of 1% Cu, and a minimum down-hole length of three metres. In the event that the assays in a drill hole could not be combined to meet the above criteria, the highest-grade composite was formed with a length similar to those of the adjacent SMZs.

The mineral resource area was divided into 10 structural domains, and a digital terrain model (dtm) was constructed through the SMZ centroids to define the geometry of the mineralization 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 mineralization.

Resources were classified using the same criteria reported in the September 5, 2012 Technical Report that require a nominal 400 m drill hole spacing for Indicated and a nominal 800 m spacing for Inferred.

AMEC 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 mineralization 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 SMZ intercepts grading a few tenths of a percent copper above and below the composite and well over 1% Cu within the SMZ composite. To test the 1% cut-off grade and various sensitivity cases for the purposes of assessing reasonable prospects of economic extraction, AMEC performed a conceptual analysis based on the metallurgical recovery algorithms, operating costs and economic parameters used in the 2012 PEA.

Two seam models were constructed below and above over the entire extent of the underground model using a 0.5% Cu cut-off to help evaluate the open-pit potential for future studies.

1.11 Mineral Resource Statement

The Mineral Resources have been defined taking into account the 2010 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, are the Qualified Persons for the Mineral Resource estimates.

Mineral Resources are stated in terms of total copper (Cu). Mineral Resources are reported at a base case total copper cut-off grade of 1% Cu and a minimum vertical thickness of 3 m and are summarised in Table 1-1.

Table 1-2 tabulates the additional Mineral Resources that may be gained if the 0.5% Cu skins adjacent to the underground resource model are mined using an open-pit mining method.

Table 1-1: Kamoa Project Indicated and Inferred Mineral Resources for Domain 1 at 1% Cu Cut-off Grade

Category	Tonnage (Mt)	Area (km^2)	Cu (%)	True Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
Indicated	739	50.5	2.67	5.20	19,700	43.5
Inferred	227	20.5	1.96	3.84	4,460	9.8

Notes:

1. Mineral Resources have an effective date of 10 December, 2012. Harry M. Parker and Gordon Seibel, both SME Registered Members, are the Qualified Persons responsible for the Mineral Resource estimates. The Mineral Resource estimate was prepared by Mr. Seibel
2. Mineral Resources are reported using a total copper (Cu) cut-off grade of 1% Cu and a minimum assumed thickness of 3 metres. A 1% Cu cut-off grade is typical of analogue deposits in Zambia. There are reasonable prospects for economic extraction under assumptions of a copper price of US$3.30/lb; sulphuric acid credits of US$300/t of acid produced; employment of underground mechanized room-and-pillar mining methods; and that copper concentrates will be produced and smelted
3. Reported Mineral Resources contain no allowances for hanging wall or footwall contact boundary loss and dilution. No mining recovery has been applied
4. The Mineral Resources include the mineralization above a 1% total copper cut-off that is potentially amenable to open-pit mining
5. Tonnage and grade measurements are in metric units. Contained copper tonnes are reported using metric units; contained copper pounds use imperial units
6. True thickness ranges from 2.4 metres to 17.6 metres for Indicated Mineral Resources and 2.8 metres to 8.4 metres for Inferred Mineral Resources
7. 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
8. Tonnages are rounded to the nearest million tonnes; grades are rounded to two decimal places
9. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content
10. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-2: Additional Resource Adjacent to Underground Model if mined in an Open Pit using a 0.5% Cut-Off

Depth Below Topo (m)	Tonnage Mt	Cu (%)	Contained Copper (kt)	Contained Copper (billion lbs)
Indicated Mineral Resources (cumulative)				
25	1.1	0.71	8	0.0
50	28	0.70	197	0.4
75	60	0.70	418	0.9
100	96	0.69	660	1.5
125	121	0.69	836	1.8
150	146	0.69	1,010	2.2
175	173	0.69	1,200	2.6
200	197	0.69	1,360	3.0
Inferred Mineral Resources (cumulative)				
Depth Below Topo (m)	**Tonnage Mt**	**Cu (%)**	**Contained Copper (kt)**	**Contained Copper (billion lbs)**
25	0.4	0.58	2	0.0
50	0.5	0.60	3	0.0
75	2	0.67	16	0.0
100	10	0.69	66	0.1
125	19	0.68	130	0.3
150	36	0.66	234	0.5
175	51	0.66	333	0.7
200	69	0.66	453	1.0

Notes:

1. Mineral Resources have an effective date of January 12, 2013. Harry M. Parker and Gordon Seibel, both SME Registered Members, are the Qualified Persons responsible for the Mineral Resource estimates. The Mineral Resource estimate was prepared by Mr. Seibel
2. Base Case of a maximum pit depth of 100 m highlighted. There are reasonable prospects for economic extraction because analogue open pits in Zambia and the DRC with similar grades and economics exist with a similar depth
3. Mineral Resources are reported using a total copper (Cu) cut-off grade of 0.5% Cu and no minimum thickness. A 0.5% Cu cut-off grade is typical of open pit deposits in the DRC
4. Tonnages are rounded to the nearest million tonnes; grades are rounded to two decimal places
5. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content
6. Tonnage and grade measurements are in metric units. Contained copper tonnes are reported using metric units; contained copper pounds use imperial units.

The Mineral Resources used for the 2012 PEA were those with an effective date of 24 September 2011 as follows:

- Total Indicated Mineral Resource estimate for the Kamoa deposit of 348 Mt grading 2.64% Cu at a cut-off grade of 1% Cu, over a minimum vertical thickness of 3 m. Based on the application of a 2% Cu cut-off grade over the same minimum thickness criteria, the Indicated Mineral Resource totals 253 Mt grading 3.02% Cu. An approximate cut-off grade of 2% was used for that subset of the Mineral Resources considered in the Preliminary Economic Assessment.
- Total Inferred Mineral Resource estimate for the Kamoa deposit of 462 Mt grading 2.72% Cu at a cut-off grade of 1% Cu, over a minimum vertical thickness of 3 m. Based on the application of a 2% Cu cut-off grade over the same minimum thickness criteria, the Inferred Mineral Resources total 332 Mt grading 3.16% Cu.

By inspection, the tonnage and grade of the material included in the 2012 PEA mine plan that is based on the September 2011 Mineral Resource estimate is not materially different to the tonnage and grade estimated for the same area in the December 2012 resource update.

1.12 Exploration Target

The area inside the model perimeter surrounding the Indicated and Inferred Mineral Resources is considered an Exploration Target. The ranges of the Exploration Target tonnages and grades are summarised in Table 1-3. Tonnage and grade ranges were estimated using an inverse distance to the fifth power for Domain 2 and applying a ±20% variance to the resulting tonnage and grade estimate.

AMEC cautions that the potential quantity and grade of 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.

1.13 Factors which may affect the Resource Estimates

Factors which may affect the Mineral Resource estimate include:

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

Table 1-3: Tonnage and Grade Ranges for Exploration Targets

Target	Low-range Tonnage Mt	High-range Tonnage Mt	Low-range Grade (%Cu)	High-range Grade (%Cu)
Total	520	790	1.6	2.5

- Faulting
  - The presence of local faulting could disrupt the productivity of a highly-mechanized operation. In addition, the amount of contact dilution related to local undulations in the SMZ has yet to be determined. Ivanplats plans to mitigate these risks with further in-fill drilling and an exploratory decline
- Metallurgical recoveries
  - Metallurgical testwork indicates the need for multiple grinding and flotation steps, and variability testwork is yet to be undertaken.
- Mining plan
  - Mining recovery could be lower and dilution increased where the dip locally increases on the flanks of the domes. An exploration decline is being considered 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. Location of an exploration decline(s) and the extent of the decline are dependent on further studies
- Infrastructure
  - Exploitation will require building a greenfields project with attendant infrastructure.
- Political setting
  - The DRC is emerging from a period of political instability, and the fiscal and political regime under which mining operations might occur are uncertain. There is provision within the DRC Mining Code for the Government to change the DRC Mining Code and mining rights by decree.

1.14 2012 Preliminary Economic Assessment

This section incorporates assumptions, analysis and findings of the Preliminary Economic Assessment that has an effective date of 5 September 2012.

The preliminary mine plan presented in this section is partly based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the Preliminary Economic Assessment based on these Mineral Resources will be realized.

The information relevant to the preliminary mine plan, prepared during the 2012 PEA, is included in this section and has not been updated because the QPs consider that the assumptions supporting the outcomes remain reasonable.

In a similar manner, the QPs' opinion is that the assumptions made with respect to infrastructure, recovery methods, estimation of capital and operating costs, marketing studies and metal price assumptions and the resulting financial analysis remain reasonable.

The effective date of the 2012 PEA results therefore remains 5 September 2012.

1.14.1 Summary of Financial Results

The plan described in the study is for the construction and operation of a long-term underground mine, concentrator processing facility and smelter operation and associated infrastructure. The mining rate and concentrator feed capacity is 5 Mtpa, producing 143 ktpa of payable copper on average in the first 10 years of operation. The production scenario schedules 299 Mt over 61 years, producing 7.8 Mt of blister copper. The production schedule includes Indicated and Inferred Mineral Resources.

The economic analysis used a long term price assumption of US$2.85/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 a Net Present Value (NPV) at a 10% discount rate of US$1.16 billion (after tax). It has an after tax internal rate of return (IRR) of 17.0% and a payback period of 5.29 years. The average total cash cost after credits in the first 10 years of production is US$0.95/lb of copper. Table 1-4 summarises the financial results, whilst Table 1-5 summarises mine production, processing and concentrate, and metal production statistics. Realisation costs are shown which indicate the actual realizable value of Payable Copper produced after accounting for the transport treatment and royalties payable on these sales.

Table 1-4: Financial Results

		Before Taxation	After Taxation
Net Present Value US$M	Undiscounted	25,430	17,640
	5.0%	6,121	4,062
	8.0%	3,091	1,929
	10.0%	2,002	1,164
	12.0%	1,286	663
	15.0%	612	196
IRR	–	20.5%	17.0%
Project Payback (years)	–	4.63	5.29

Table 1-5: Mining and Processing Production Statistics

	Total LOM	First 5 Years Average	First 10 Years Average	LOM Average
Total Plant Feed Mined ('000 t)	299,391	4,463	4,731	4,908
Quantity Plant Feed Treated ('000 t)	299,388	4,800	4,900	4,908
Copper Feed Grade (%)	3.14	3.29	3.50	–
Copper Recovery (%)	84.8%	84.3%	85.3%	–
Concentrate Produced ('000 t)	24,384	411	450	400
Copper Concentrate Grade (%)	32.7%	32.3%	32.6%	–
Contained Metal in Concentrate				
Copper ('000t)	7,984	133	147	131
Copper (Mlb)	17,601	293	324	289
Payable Copper				
Copper ('000t)	7,801	130	143	128
Copper (Mlb)	17,198	287	316	282

Note: First 5 Years Average and First 10 Years Average plant feed mined does not include mined material prior to Year 1 (1,688 kt).

Figure 1-1 and Figure 1-2 depict the processing, concentrate and metal production, respectively.

Figure 1-1: Processing Production

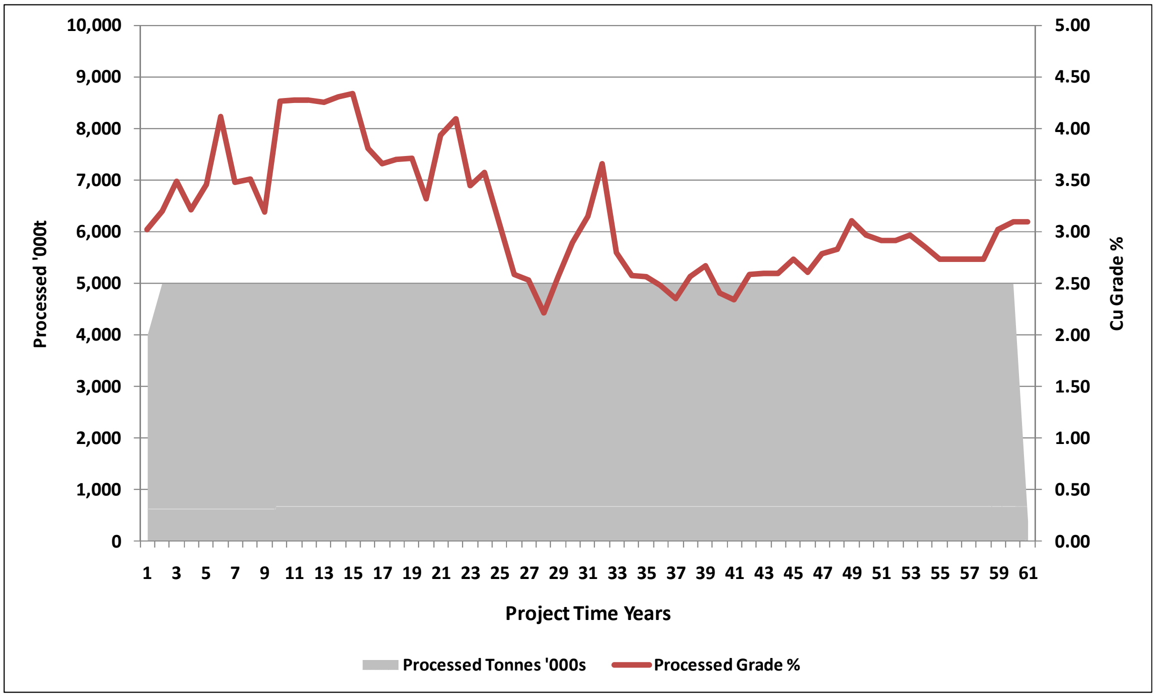


Figure 1-2: Concentrate and Metal Production

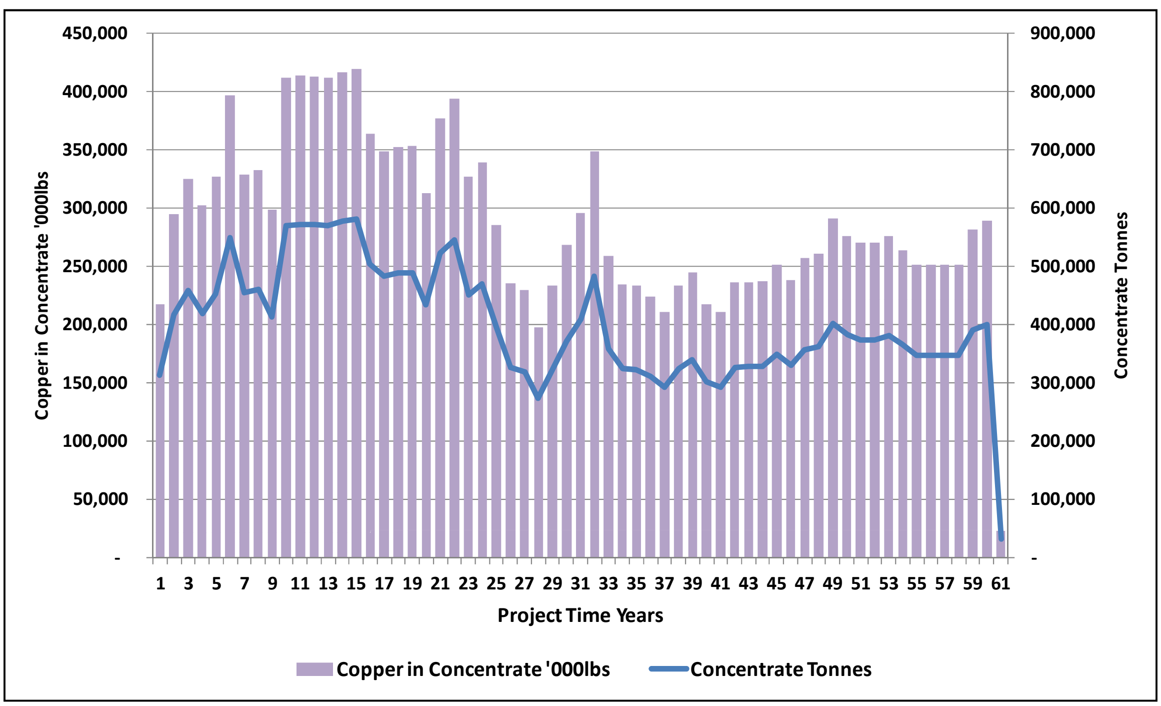


Table 1-6 summarises unit operating costs, whilst Table 1-7 provides a complete breakdown of operating costs and revenue.

Table 1-6: Unit Operating Costs

	US$/lb Payable Copper		
	LOM Average	**First 5 Years Average**	**First 10 Years Average**
Mine Site Cash Cost	1.04	0.93	0.89
Realisation Cost	0.30	0.30	0.30
Total Cash Costs Before Credits	**1.34**	**1.22**	**1.19**
Acid Credits	0.24	0.24	0.24
Total Cash Costs After Credits	**1.10**	**0.99**	**0.95**

Table 1-7: Operating Costs and Revenues

	US$M	US$/t Plant Feed Milled		
	Total LOM	**First 5 Years**	**First 10 Years**	**LOM Average**
Revenue				
Copper in Blister	49,013	170.16	183.85	163.71
Acid	4,047	14.23	15.26	13.52
Gross Sales Revenue	**53,061**	**184.39**	**199.11**	**177.23**
Less: Realisation Costs				
Transport	2,988	10.37	11.21	9.98
Treatment and Refining	786	2.73	2.95	2.63
Royalties and Export Tax	1,357	4.71	5.09	4.53
Total Realisation Costs	**5,132**	**17.82**	**19.25**	**17.14**
Net Sales Revenue	**47,929**	**166.57**	**179.86**	**160.09**
Site Operating Costs				
OP & UG Mining	10,476	28.92	32.20	34.99
Processing	3,125	10.55	10.45	10.44
Smelting	2,203	7.65	7.90	7.36
Tailings	83	0.27	0.27	0.28
General & Administration	1,616	6.87	5.63	5.40
Customs	342	1.01	1.09	1.14
Total	**17,844**	**55.28**	**57.54**	**59.60**
Operating Margin	**30,085**	**111.29**	**122.32**	**100.49**

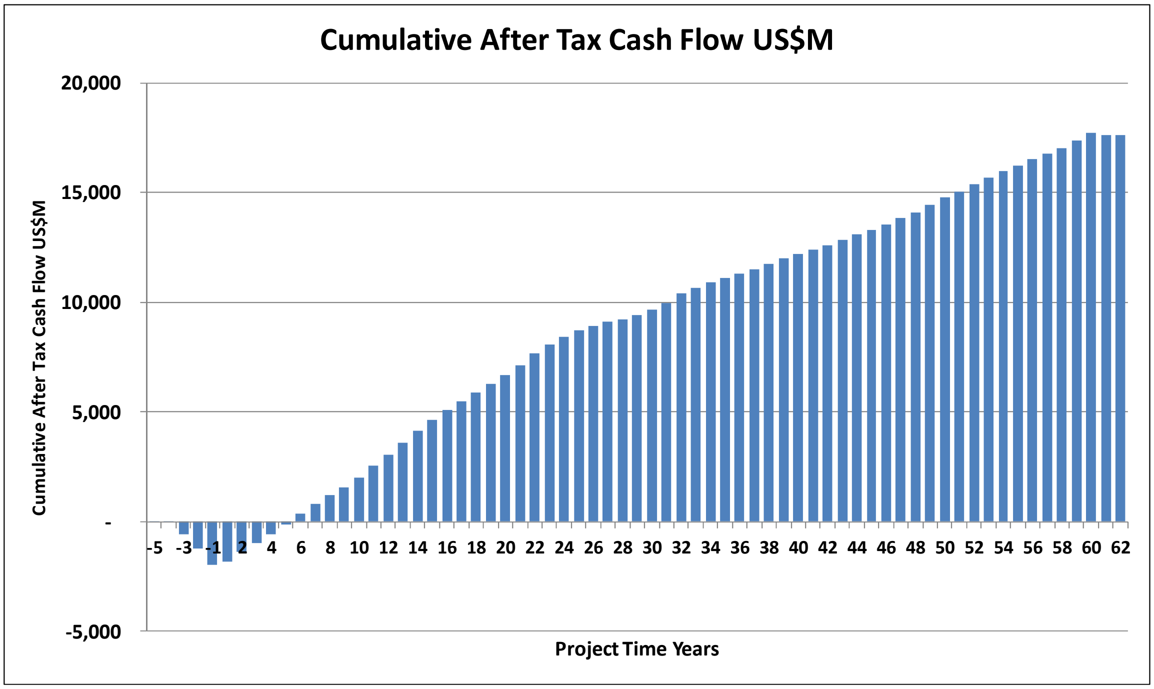
The capital costs for the project are detailed in Table 1-8.

Table 1-8: Capital Investment Summary

US$M	Phase 1	Sustaining	Total
Mining			
Open Pit Mining	63	5	67
Underground Mining	173	1,146	1,319
Subtotal	**235**	**1,151**	**1,386**
Power & Smelter			
Smelter	313	38	351
Power	141	85	226
Subtotal	**455**	**122**	**577**
Concentrate & Tailings			
Concentrator	270	12	281
Tailings	42	372	414
Subtotal	**312**	**383**	**696**
Infrastructure			
Site Facilities and Temporary Works	234	141	375
Off-site Infrastructure	24	14	38
Subtotal	**257**	**155**	**413**
Indirects			
Construction Indirects EPCM	155	2	157
Subtotal	**155**	**2**	**157**
Owners Cost (incl. Drilling & Studies)			
Owners Cost	205	92	298
Closure		200	200
Subtotal	**205**	**292**	**498**
Capital Expenditure Before Contingency	**1,621**	**2,106**	**3,727**
Contingency	368	453	821
Capital Expenditure After Contingency	**1,989**	**2,560**	**4,549**

Cumulative after tax cash flow is depicted in Figure 1-3.

Figure 1-3: Cumulative Cash Flow



1.15 Conclusions

1.15.1 Mineral Resource Estimate Update

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 (2010).

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-mechanized operation. In addition, the amount of contact dilution related to local undulations in the SMZ has yet to be determined. Ivanplats plans to mitigate these risks with further in-fill drilling and an exploratory decline
- Assumptions used to generate the data for consideration of reasonable prospects of economic extraction. Mining recovery could be lower and dilution increased where the dip locally increases on the flanks of the domes. An exploration decline is being considered 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. Location of an exploration decline(s) and the extent of the decline are dependent on further studies

- Long-term commodity price assumptions
- Long-term exchange rate assumptions
- Operating and capital assumptions. Exploitation will require building a greenfields project with attendant infrastructure
- Metal recovery assumptions. Metallurgical testwork indicates the need for multiple grinding and flotation steps, and variability testwork is yet to be undertaken
- The fiscal and political regime under which mining operations might occur are uncertain. There is provision within the 2002 Mining Code for the Government to change the 2002 Mining Code and mining rights by decree and a draft 2012 Mining Code has been circulated. There is also a risk that the DRC Government could change the current royalty, duty, and taxation regime.

1.15.2 2012 PEA

As the 2012 PEA is still considered to be current, the following conclusions are considered to remain reasonable as their underlying assumptions have not changed.

The 2012 PEA examined an initial 5 Mtpa mine production rate without expansions over the mine life. Further study work should be undertaken to optimise the project production rate by considering concentrator and smelter capacities that are matched to the mine production and the available power supply.

The key mine production criterion used for production scheduling in the 2012 PEA was the concentrator feed rate and this resulted in variable concentrate production. Further work should be carried out using a concentrate production rate suited to the smelter capacity. This will allow for a more efficient use of the smelter capital.

Given the significant Mineral Resource tonnage estimated and its large lateral extent, potential mining rates could range from 5 Mtpa to 20 Mtpa, through operating in multiple mining areas and a series of production expansions to maximise extraction of the Mineral Resource.

The 2012 PEA includes capital for an upgrade of the power generating capacity in the region. The power consumption assumed for the 5 Mtpa option in this report does not require the full generating capacity that would result from the capital expenditure. Preliminary work indicates that a concentrator capacity of 7.5 Mtpa may allow a more efficient use of the assumed capital and it is recommended that further study be undertaken to examine this rate as the revised base case capacity.

Selection of the initial plant feed rate to suit the available power capacity may maximise the financial returns from the project.

It is recommended that a further round of study include trade-off analysis of capacity and capital to identify additional optimisations for the Project’s production rate.

Ivanplats has undertaken some initial power studies that have identified the availability of initial power capacity. Kamoa contains a large Mineral Resource and production rate studies have been recommended. Ivanplats will continue to study the power capacity and generation capabilities in the region and use these results for defining production capacity. After developing an understanding of the power options in the region Ivanplats may wish to consider participating in other power projects to secure power for later expansions. Ivanplats have already started in implementing this strategy after having signed an MOU with the Ministry of Energy for the identification of potential sites for a greenfield power plant and have conducted a due diligence operation on available sites to identify a suitable site for this purpose.

Matching of the mining, concentrate production and infrastructure capabilities will require consideration of the ramp up rates and full production rates. For example, relatively small amounts of open pit production are required for the plant feed to allow an earlier start than would be possible from an underground only operation. The work on the open pit mineral resource should be continued as well as options for the declines and underground production rates. The geotechnical mine design criteria for underground mine design, which impacts resource recovery, dilution and mining costs, are mainly based on widely spaced drill holes and for more detailed study this will need to be confirmed by data to be obtained from the underground exposure. Therefore, the mine designs will need to be revised according to the geotechnical recommendations.

1.16 Recommendations

As an update to the 2012 PEA is underway, AMEC has restricted recommendations to a work program of one phase of drilling. The recommended drilling has been broken down by localities within the deposit, and totals 85,231 m, including allocations for exploration, infill, metallurgical, geotechnical, and condemnation purposes. The program is estimated at US$46.5 M.

2.0 INTRODUCTION

AMEC E&C Services Inc (AMEC) has prepared an independent Technical Report (the Report) on the Kamoa Copper Project (the Project) located in the Democratic Republic of Congo (DRC) for Ivanplats Limited (Ivanplats). The project location is outlined in Figure 2-1.

The Report documents a Mineral Resource estimate update and also includes a summary of the Preliminary Economic Assessment study completed on the Project with an effective date of 5 September 2012 (2012 PEA). Results from the 2012 PEA mining studies have not changed in terms of their outcomes as their underlying assumptions remain reasonable.

In addition to AMEC, the following companies were involved in the preparation of the 2012 PEA:

- AMC Consultants: Overall report preparation, financial model
- Stantec Inc.: Underground mine plan
- Golder Associates: Hydrogeology and water
- African Mining Consultants: Environmental
- SRK Consulting Inc.: Underground geotechnical recommendations.

2.1 Terms of Reference

The Report was prepared in support of Ivanplats' Annual Information Form for the year ended 31 December, 2012.

The Project is 95% held in the name of African Minerals (Barbados) Ltd. sprl (AMBL), a wholly-owned Ivanplats subsidiary. For the purposes of this report, the name "Ivanplats" refers interchangeably to Ivanplats' predecessor company, Ivanhoe Nickel and Platinum Ltd., and the current subsidiary companies. The remaining 5% Project interest is held by the Government of the DRC.

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

Figure 2-1: Project Location Map

Note: Figure courtesy Ivanplats, 2011

2.2 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:

- Dr Harry Parker, Registered Member, Society for Mining, Metallurgy and Exploration (RM SME), Membership Number (#) 2460450, Technical Director, AMEC
- Mr Gordon Seibel, RM SME, #2894840, Principal Geologist, AMEC
- Mr Dean David, Fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM), #102351, Technical Director Process, AMEC
- Mr Bernard Peters, FAusIMM #201743, Mining Manager, AMC Consultants

- Mr Jarek Jakubec, Registered Chartered Engineer (C. Eng.) #509147, and Member of the Institute of Materials, Minerals and Mining of the United Kingdom (MIMMM), Corporate Consultant, SRK Consulting Inc
- Mr Mel Lawson, RM SME, # 1859650, Mining Principal, Stantec Mining.

2.3 Site Visits and Scope of Personal Inspection

Site visits were performed as follows.

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 Ivanplats 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 mineralization interpretations with Ivanplats' staff.

Mr Gordon Seibel visited the Project from 9 to 10 February, 2011, from 5 to 8 November 2011, and again from 12–14 November, 2012. 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 mineralization interpretations with Ivanplats' staff, focusing on deposit strike, dip, and faulting geometries.

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

Mr Bernard Peters visited the site from 15 February 2010 to 17 February 2010 and again 27 to 30 April 2010; and again on 15 November 2012. The site visits included briefings from Ivanplats 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.

Mr Jarek Jakubec visited the Kamoa Project from 27 to 30 April 2010. The site visit included attending presentations by Ivanplats and AMEC staff, inspection of core and surface outcrops, review of the geotechnical logging procedures, and point load testing and structural interpretations with Ivanplats' staff.

Mr Mel Lawson visited the Kamoa Project on 15 November 2012. During the site visit, Mr Lawson attended a briefing by local staff, undertook a camp tour, drill core inspection (both general and from the proposed Centrale portal), and conducted a general site reconnaissance driving tour and inspection of the proposed Centrale portal site. Mr Lawson also attended a Kamoa Project kick-off meeting in Johannesburg from 12–13 November, 2012, prior to visiting site.

2.4 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: 11 March 2013
- Date of database close-out date for updated Mineral Resource estimate: 10 December 2012
- Date of database close-out date for estimation of Mineral Resources that support the 2012 PEA: 24 September 2011
- Date of Mineral Resource update for mineralization amenable to underground mining methods: 10 December 2012
- Date of Mineral Resource estimate that supports the 2012 PEA: 24 September 2011
- Date of the Mineral Resource estimate for mineralization amenable to open-pit mining methods: 12 January 2012
- Date of supply of mineral tenure and surface rights information for updated Mineral Resource estimate: 5 September 2012
- Date of the supply of legal information supporting mineral tenure that supports the 2012 PEA: 5 September 2012
- Date of the final information supporting the financial analysis in the 2012 PEA: 5 September 2012.

The effective date of the Report is taken to be the effective date of the last information supplied on the ongoing drilling program and is 11 March 2013.

2.5 Information Sources and References

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

2.6 Previous Technical Reports

Ivanplats have previously filed a technical report on the Project, entitled:

Peters, B.F, Parker, H.M., Seibel, G., Jakubek, J., Annavarapu, S., and David, D., 2012: Ivanplats Limited, Kamoa Project, Katanga Province, D.R.C., NI 43-101 Technical Report: unpublished technical report prepared for Ivanplats Limited by AMC Consultants, AMEC E&C Services Inc., SRK Consulting Inc and Stantec Inc., effective date 5 September 2012, 375 p.

3.0 RELIANCE ON OTHER EXPERTS

The AMEC QPs, as authors of this Report, have relied on, and believe there is a reasonable basis for this reliance, upon the following Other Expert reports as noted below.

3.1 Mineral Tenure

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

- Cabemery and Partners Pty Ltd., 2012: Validity Of (I) Exploration Permits Relating To The Mining Project Of Kamoa Held By The Company African Minerals (Barbados) Ltd Sprl, (Ii) Their Renewal And (Iii) Transformation Of Some Of Them Into Exploitation Permits, addressed to Ivanplats Limited, BMO Nesbitt Burns Inc., Morgan Stanley Canada Limited and AMC Consultants Pty Ltd, 5 September 2012.

This information was used in Sections 4.3 and 4.4 of the Report.

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

- Broughton, D., 2010: Kamoa: unpublished internal email to AMEC, dated 24 August 2010.
- Geraghty, L., 2010: PR 702 to 706: Copies of Original Signed Permit Certificates: unpublished internal email to AMEC, dated 8 September 2010.
- Broughton, D., 2012: Kamoa Project: unpublished letter prepared by representatives of Ivanhoe Nickel and Platinum Limited for AMC and AMEC, received 12 June 2012.

This information was used in Sections 4.3 and 4.4 of the Report.

3.2 Surface Rights

The AMEC QPs have fully relied upon, information supplied by Ivanplats' 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:

Cabemery and Partners Pty Ltd., 2012: Validity Of (I) Exploration Permits Relating To The Mining Project Of Kamoa Held By The Company African Minerals (Barbados) Ltd Sprl, (Ii) Their Renewal And (Iii) Transformation Of Some Of Them Into Exploitation Permits, addressed to Ivanplats Limited, BMO Nesbitt Burns Inc., Morgan Stanley Canada Limited and AMC Consultants Pty Ltd, 5 September 2012.

This information was used in Section 4.3 and 4.4 of the Report.

The AMEC QPs have fully relied upon, information supplied by Ivanplats staff and experts retained by Ivanplats 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, 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.
- Broughton, D., 2012: Kamoa Project: unpublished letter prepared by representatives of Ivanhoe Nickel and Platinum Limited for AMC and AMEC, received 12 March 2012.

This information was used in Section 4.3 and 4.4 of the Report.

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 Ivanplats, and from information supplied by Ivanplats staff. The AMEC QPs have fully relied upon information derived from such experts through the following documents:

- Broughton, D., 2010: Kamoa: unpublished internal email to AMEC, dated 24 August 2010.

This information was used in Section 4.8 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.

3.4 Taxation and Royalties

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

- KPMG Services Proprietary Limited, 2012: Letter from M Saloojee to M Cloete regarding high-level commentary on the tax assumptions and tax rates applicable to an operating mine in the Democratic Republic of Congo, dated 13 January 2012.
- Broughton, D., 2012: Kamoa Project: unpublished letter prepared by representatives of Ivanhoe Nickel and Platinum Limited for AMC and AMEC, received 12 March 2012.

This information was used in Section 22 of the Report.

4.0 PROPERTY DESCRIPTION AND LOCATION

The Kamoa Project is situated in the Kolwezi District of Katanga 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 2-1).

The Project is centred at approximate latitude 10°46'S and longitude 25°15'E.

4.1 Property and Title in the Democratic Republic of Congo

4.1.1 Introduction

A summary of the mining history of the Katangan region is presented below, and is adapted from Goossens (2009), King (2009), and André-Dumont (2008, 2011).

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.

When the 2002 Mining Code was introduced, the DRC Government indicated that after a 10-year period, a review would be undertaken. In late 2012, information on draft changes were released, and the wording of the draft changes are incorporated in the following subsections where the current code has the potential to be amended. Media commentators have noted that a new mining law may be expected during 2013 (Reuters, 2012).

4.1.2 Mineral Property Title

The following summary on mineral title is adapted from André-Dumont (2008, 2011), 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. Key players in the administration of mining activities 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.

AMEC notes that 2012 draft changes to the 2002 Mining Code suggest that the DRC Government is considering increasing the 5% non-dilutable interest to a 35% stake in projects that is "free of charges and non-dilutable" (Reuters, 2012).

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.

4.1.3 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. AMEC notes that 2012 draft changes to the 2002 Mining Code would reduce the permit duration to three years (Reuters, 2012).

No individual Exploration Permit can exceed a surface area of 400 km². 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 km².

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.

4.1.4 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.

AMEC notes that 2012 draft changes to the 2002 Mining Code would reduce the permit duration from 30 years to 25 years (Reuters, 2012).

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:

[1] EIS = Environmental Impact Statement

[2] EMPP = Environmental Management Protection Plan

"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
- 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”.

4.1.5 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 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.

4.1.6 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.

4.1.7 Royalties

According to the 2002 Mining Code 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 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. Different rates apply to different types of metals sold.

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

AMEC notes that 2012 draft changes to the 2002 Mining Code include a proposal to double royalties on some minerals and introduces a 50% levy on miners' "super profits" (Reuters, 2012). According to the draft, mining royalties on non-ferrous metals will increase to 4% from 2%. They will also increase to 6% from 2.5% for strategic and precious metals, and to 6% from 4% on precious and coloured stones. The draft changes define "super profits" as those made when a commodity's price rises exceptionally over 25% compared with its level at the time of the project's feasibility study (Reuters, 2012).

4.1.8 Key Taxes

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.

Mining companies are subject to tax on rental income, on movable income and corporate income. Tax on movable income is levied at a rate of 20%. Some exemptions are available:

- An exemption for interest paid on a loan in a foreign currency
- A reduction to 10% in the rate payable on dividends.

Corporate tax is levied at 30% of income, increasing to 40% if the product is refined or smelted offshore.

Vehicle, real estate, mining and hydrocarbon concession areas taxes are payable to the tax authority of the province where the owner of mining rights carries out its mining activities.

In 2012 the DRC adopted a VAT regime; the standard VAT rate is 16% levied on all supplies of goods and services rendered and is not levied on any capital asset movements (Ivanplats, 2012).

Employee tax takes two forms:

- A graduated withholding tax on all forms of employee income which varies from 3% to 50%
- An additional 10% tax on expatriate employees, payable by the employer.

Mining companies are subject to import duties on all goods and products imported in accordance with a preferential customs regime.

AMEC notes that the Katanga Province has imposed a provincial tax on mining concentrate products destined for export, of up to $60 per tonne (Ivanplats, 2012).

4.2 Project Ownership

The Project is 95% held in the name of African Minerals (Barbados) Ltd. sprl (AMBL), a wholly-owned Ivanplats subsidiary. On 11 September, 2012, Ivanplats transferred a 5% non-dilutable interest in the share capital of AMBL to the to a DRC state-owned nominee. The transfer was triggered by the grant of the Kamoa Exploitation Licences on August 20, 2012, whereby the 5% interest had to be transferred to the DRC Government within a 30 day period.

Ivanplats has offered to sell the DRC Government an additional 15% interest in the Project on commercial terms to be negotiated.

4.3 Mineral Tenure

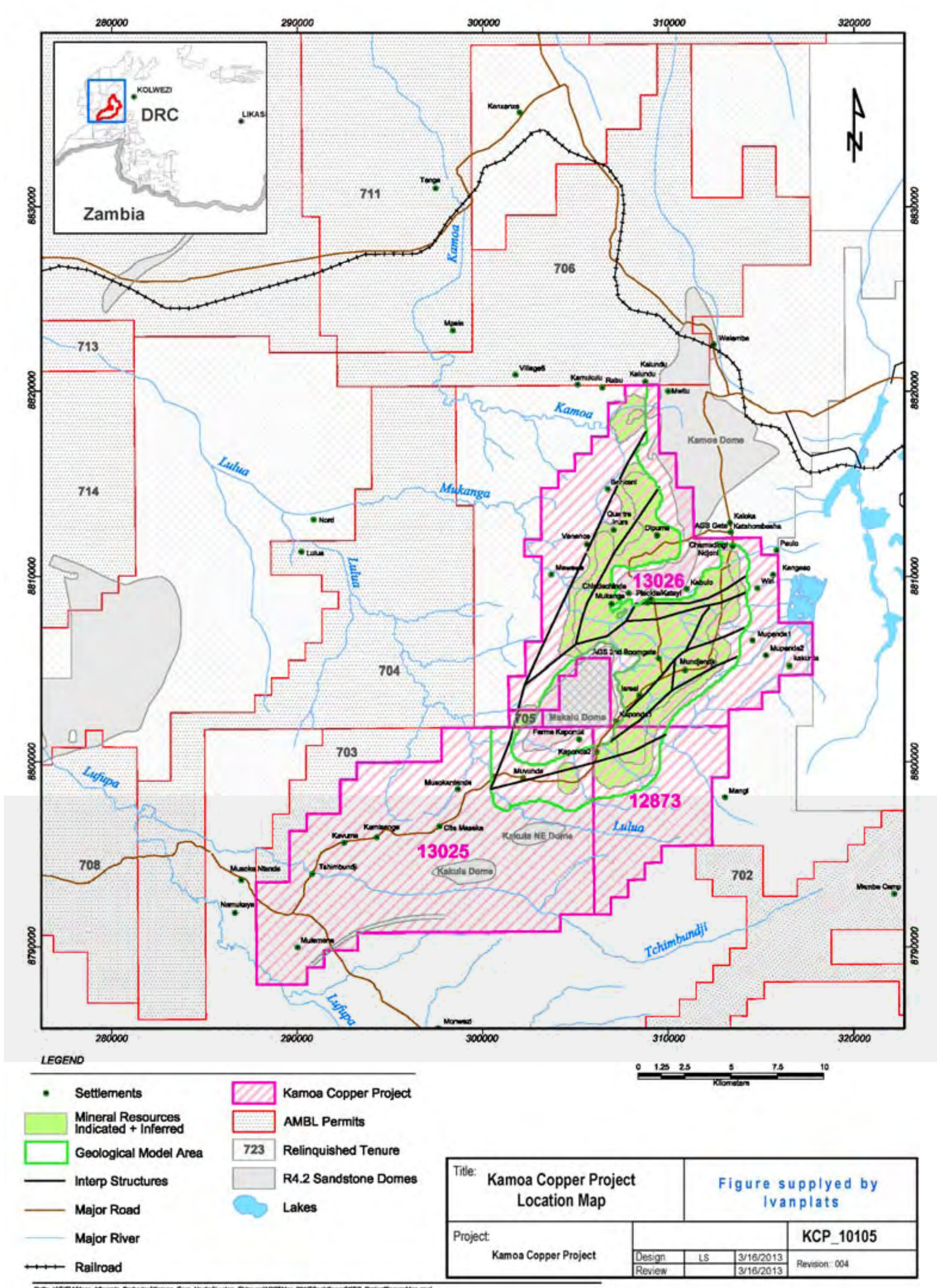
The Project consists of three granted Exploitation Permits 12873, 13025 and 13026 and one Exploration Permit, 705 (Table 4-1 and Figure 4-1). The permits are held in the name of AMBL. The three Exploitation Permits cover an area of 397.4 km^2, and the Exploration Permit covers 1.7 km^2 for a total Project area of 399.1 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 (km^2)
12873	20 Aug 2012	19 August 2042	Silver, Bismuth, Cadmium, Cobalt, Copper, Iron, Germanium, Nickel, Gold, Palladium, Platinum, Lead, Rhenium, Sulphur and Zinc.	62	52.7
13025	11 Nov 2003	19 August 2042	Silver, Bismuth, Cadmium, Cobalt, Copper, Iron, Germanium, Nickel, Gold, Palladium, Platinum, Lead, Rhenium, Sulphur and Zinc.	204	173.2
13026	11 Nov 2003	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 (km^2)**
705	11 Nov 2003	10 May 2015	Base, Precious, Platinum Group Metals, Pegmatite Minerals, Diamonds and Gemstones	2	1.7
Sub Total					1.7
TOTAL					**399.1**

Figure 4-1: Project Tenure Plan



Note: Figure courtesy Ivanplats, modified by AMEC, 2013

Ivanplats advised the AMEC 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. Ivanplats advised the QPs that the required land tax payments for 2013 were made for the three Exploitation Permits and Exploration Permit 705, and that the company expects to pay the surface rights fee required for the permits on or before 31 March 2013, the due date.

Ivanplats 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, Ivanplats 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 Ivanplats.

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

4.5 Royalties and Encumbrances

A discussion of the royalty considerations payable to the DRC Government is included in Section 4.1.8.

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 Significant Risk Factors

As noted in Section 4.2, the DRC Government is currently considering an update to the 2002 Mining Code. Legislation is currently in draft form. The most significant impact on the Project would be if the DRC Government holdings is mandated as a 35% stake that is "free of charges and non-dilutable" (Reuters, 2012). Additional Project impacts may occur if there is a percentage increase in the royalty regime, incorporation of a super profits tax, and if the Katangan province maintains or increases the provincial tax on mining concentrate products destined for export, of up to $60 per tonne.

4.11 Comments on Section 4

In the opinion of the AMEC QPs, the information discussed in this section supports the declaration of Mineral Resources:

- Information from legal experts supports that the mining tenure held is valid and is sufficient to support declaration of Mineral Resources
- Surface rights have yet to be obtained. The Project area is sparsely inhabited, and Ivanplats' investigations to date have identified only one surface rights holder; however, some additional surface rights within the area of the Project may be expected to be held by private individuals
- Ivanplats will need to apply for additional specialist permits as appropriate to the jurisdiction to allow mining operations
- Ivanplats has offered to sell the DRC Government an additional 15% interest in the Project on commercial terms to be negotiated.
- A draft 2012 Mining Code to replace the existing 2002 Mining Code has been circulated, but not promulgated; under this draft Mining Code, the DRC Government is entitled to a 35% project interest
- 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.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Accessibility

5.1.1 Air

The city of Lubumbashi, 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 southeast of Lubumbashi, can be used.

The closest major township to the Project is Kolwezi 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 Likasi and Lubumbashi by road. Travel time by car from Kolwezi to Lubumbashi is currently five to six 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 unsealed roads to the villages of Kasekelesa and Musokantanda. The road network throughout the Project has been upgraded by Ivanplats to provide reliable drill and logistical access.

5.1.3 Rail

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 12 km from the central part of the area where Mineral Resources have been estimated.

5.2 Climate

The Kolwezi area has distinct dry (May to October) and wet (November to April) seasons.

Mean annual rainfall, based on data collected at the Kolwezi meteorological station varies between 870 and 1,420 mm. The major precipitation events are wet-season thunderstorms.

A hydrological monitoring program has commenced to collect site-specific data.

Temperatures are generally mild and vary between 17°C and 26°C, but can drop to as low as 5°C during the night in the winter months of July and August.

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, Ivanplats successfully conducted exploration programs on a year-round basis in 2009 and again in 2011.

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. Ivanplats 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 utilized by the Project.

The infrastructure requirements envisaged in the 2012 PEA are discussed in Section 18.

5.4 Physiography

The Project lies at an altitude of approximately 1,430 m above mean sea level. The topography is gently undulating with a few highlands (broad domes).

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

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

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

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

5.5 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 Ivanplats.

In the opinion of the QPs, the access, physiography, local services, plus existing and planned infrastructure can support the declaration of Mineral Resources.

It is expected that any future mining operations will be able to be conducted year-round.

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

A first-time Mineral Resource estimate was prepared by AMEC for the Project in 2009 under the direction of Dr. Harry Parker (Parker, 2009), and this was updated in 2010 (Parker et al., 2010). A third update was prepared in early 2011 (Parker et al., 2011), and the estimate that supports the 2012 PEA discussed in Sections 16 to 22 was completed in September 2011 (Peters et al., 2012). The current mineral resource estimates discussed in Section 14 are an update of the estimate that supports the 2012 PEA.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

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

7.1 Regional Geology

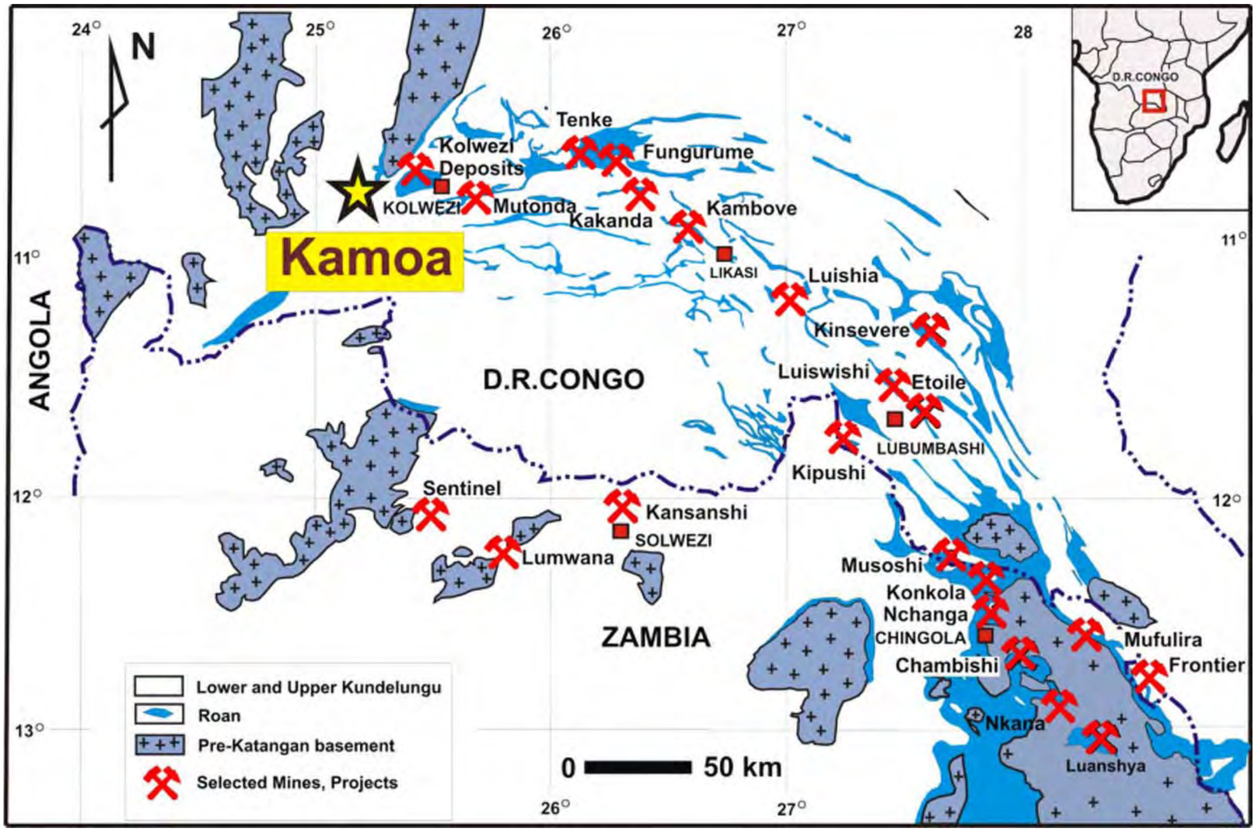
The regional geological setting for the Project is indicated in Figure 7-1.

The metallogenic province of the Central African Copperbelt is hosted in meta-sedimentary rocks of the Neoproterozoic Katanga Basin. 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 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 Project area in western Katanga, only Mesoproterozoic basement (~1,100–1,300 Ma) is known. Rainaud et al., (2005) have presented evidence for older (unexposed) Archean basement in the DRC segment.

The 5–10 km thick Katanga Supergroup in the DRC sector is traditionally subdivided into the Roan (code R), Lower Kundelungu or Kundelungu Inferieur code (Ki), and Upper Kundelungu or Kundelungu Superieur (Ks) Groups, based on two regionally extensive diamictites. Table 7-1 summarizes the typical stratigraphic sequence for the DRC sector.

Figure 7-1: Geological Setting Central African Copperbelt



Note: Figure courtesy Ivanplats, 2012

Table 7-1: Stratigraphic Sequence, Congolese (Katangang) Copperbelt

Group	Subgroup	Formation	Member	Lithology	Mineralization
Ks: Upper Kundelungu	Plateaux	Ks3		Arkoses, sandstones.	
	Ks2: Kiubo	Ks2.1/2.2		Similar to Ks1.3, but with increasing sandstone, sandy shale and siltstone	
	Ks1: Kalule	Ks1.2/1.3		Alternating sandstone and shale at top, Ks1.3; micaceous and calcareous shales; dolomitic limestone at base.	
		Ks2.1.1	Petit Conglomerat	Tillite/diamictite.	
Ki: Lower Kundelungu	Ki2: Monwezi	Ki2.1/2.2.		Coarse sandstone (greywacke); calcareous shale towards south.	
	Ki1: Likasi	Ki1.3		Argillaceous sandstone (calcareous); minor siltstone.	
		Ki1.2.2	Kakontwe	Dolomitic limestone in south.	
		Ki1.2		Alternating finely bedded shales, sandy shale and dolomite at top.	
		Ki1.1	Grand Conglomerat	Tillite; minor sandstone; interbedded pyritic sandy siltstone (KPS).	Kamoa
R: Roan	R4: Mwashya	R4.2		Bedded, weakly calcareous and siliceous shale; locally thin graphitic shales; feldspathic sandstone on basin margin.	
		R4.1		Ferruginous dolomite; ironstone with minor jaspilite overlying variably silicified dolomite, and dolomitic shale. Local volcaniclastic rocks.	
	R3: Dipeta	R3.3/3.4		Dolostones alternating with shaley micro-sandstone; minor limestone.	
		R3.2		Dolomitic evaporitic shales and sandstone; silicified dolomite towards top.	
		R3.1.	RGS	Violet–grey dolomitic shale with grit; sandy at base and top.	
	R2: Mines	R2.3: Kambove Dolomite (CMN)	Upper	Pink–brown–white dolomite; talcose, cherty, evaporitic breccia; red siltstone.	
				Dolomite (stromatolitic); talcose dolomite; evaporitic breccia; pale grey–green siltstone.	
				Pink–brown–white massive dolomite.	
			Lower	Dolomitic shale alternating with shaley	

Group	Subgroup	Formation	Member	Lithology	Mineralization
				dolomite, locally talcose, rarely sandy. Massive and bedded with algal structures.	
				Laminated carbonaceous shaley. Dolomite with evaporitic structures; minor massive dolomite.	
		R2.2: Dolomitic Shale (SD)	SD3b	Carbonaceous shale.	
			SD3a	Dolomitic shale, shaley dolomite.	
			SD2d	Carbonaceous dolomitic shale.	
			SD2c	Dolomitic shale; dolomite.	
			SD2b + c	Shaley and local sandy dolomite; dolomitic sandstone.	
			SD2a	Carbonaceous dolomitic shale.	
			BOMZ	Black Ore Mineralized Zone dolomite.	Tenke–Fungurume
			SDB	Basal dolomitic shale.	” “
		R2.1: Kamoto Dolomite	RSC	Vuggy silicified dolomite.	” “
			RSF	Laminated silicified shale.	” “
			DSTRAT	Bedded dolomite.	“ “
			RAT Gris	Grey–green argillite (rarely sandy).	“ “
	R1: RAT	RAT Lilas		Sandy (dolomitic) argillite and argillaceous sandstones.	

Note: Tenke–Fungurume is held by third-parties and is not part of the Kamoa Project. The deposit is included as an example of the more typical stratigraphic setting of the known écailles-style copper mineralization in the DRC Copperbelt.

Nomenclature has informally been revised to the Roan, N’Guba (code Ng; formerly Lower Kundelungu) and Kundelungu (code Ku) Groups. However, geological and lithological descriptions in this Report use the traditional nomenclature.

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.

The tectonic setting of the Roan Group is somewhat uncertain due to orogenic overprinting. The most common interpretation has Roan Group sediments accumulating 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).

Mineralization in the majority of the Katangan Copperbelt orebodies such as at Kolwezi and Tenke–Fungurume (refer Figure 7-1) is hosted in the Mines supergroup (R2). The mineralization 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.

7.1.1 Lufilian Orogeny

The Katangan basin was inverted[3] 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.

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.

At least two periods of magmatism accompanied deposition of the Katanga Supergroup. A period of extensional mafic magmatism, broadly co-eval with deposition of the Mwashya Subgroup and Lower Kundelungu Group sediments,

[3] Basin inversion is the process of shortening an extensional sedimentary basin whereby the basin fill is uplifted and partially extruded, and pre-existing faults are re-activated.

comprises a series of (alkaline) gabbroic intrusions and mafic volcanism dated at around 750 Ma (Selley et al., 2005).

7.2 Project Geology

The Project lies within the interpreted extension of the Western Foreland unit of north-western Zambia (Key et al., 2001). In the Project area, the basement is Kibaran Group metasedimentary rocks. Figure 7-2 summarizes the Project geological setting.

Geological mapping and magnetic data indicate that the majority of the Foreland located west of Kolwezi in the Ivanplats permits is underlain at surface by the Grand Conglomerat diamictite (Ki1.1). This unit constitutes a regional exploration target, as the base of the diamictite is considered a regionally-prospective redox boundary. The Project comprises the eastern part of this target area, where Ivanplats has focused exploration, and has discovered Zambian Copperbelt-style copper mineralization.

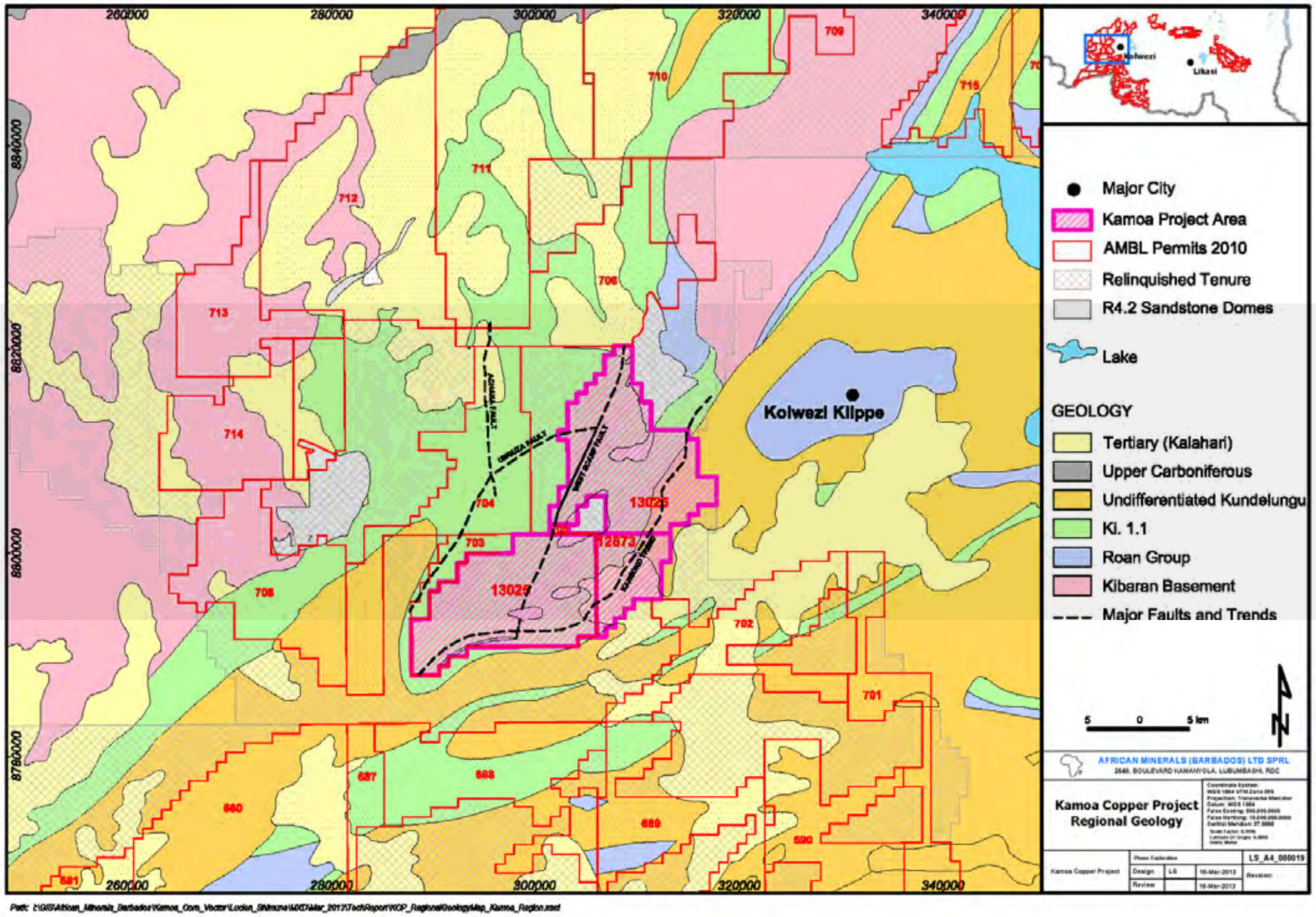
The majority of the Project area, and the area containing the Indicated and Inferred Mineral Resources discussed in Section 14 of this Report, lie on a broad, gentle plateau between two major north–northeast-trending structures. To the east, and identified primarily by airborne magnetics, is the Kansoko 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.

Three major structural blocks have been identified in the Kamoa area:

- A deep graben (or half graben) to the west of the West Scarp Fault. This is in-filled by Lower Kundelungu diamictite (Grand Conglomerat) (Ki1.1) and associated sediments which probably rest unconformably upon andesitic/mafic igneous and volcanic rocks in parts of Exploration Permit 704 (outside the Project area). The downward displacement of the West Scarp Fault is approximately 350 m to 400 m.
- An area between the West Scarp Fault and the Kansoko Trend, where the domes occur. The domes comprise Roan-age, medium- to coarse-grained feldspathic sandstone and siliciclastic rocks (footwall feldspathic sandstone–RFS) which in the north pass downwards into pebbly immature grits and conglomerates.

Figure 7-2: Regional Geology Map, Kamoa Region



Note: Figure courtesy Ivanplats, modified by AMEC, 2013

- A north–northeast to south–southwest-trending belt, the Kansoko Trend, approximately 3 km wide commencing approximately 1 km to 2 km east of the Kamoa and Makalu domes. This belt is underlain by a thick, easterly-dipping sequence of weakly carbonaceous and pyrrhotite-bearing diamictite and siltstone (Ki1.1) with subordinate andesitic (or mafic), sill-like bodies towards the northeast.

In addition to these structures, several other faults have been interpreted within the Project (Figure 7-3).

7.3 Deposits

The Project is located in a broadly-folded terrane centred on the Kamoa and Makalu domes (refer to Figure 7-3) between the West Scarp Fault and Kansoko Trend. The domes form erosional windows exposing the redox boundary between the underlying hematitic (oxidised) Roan sandstones (red-beds), and the overlying carbonaceous and sulphidic (reduced) Grand Conglomerat diamictite (host to mineralization). 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.

7.3.1 Lithology

The main stratigraphic units encountered in the Kamoa Project are summarized in Table 7-2.

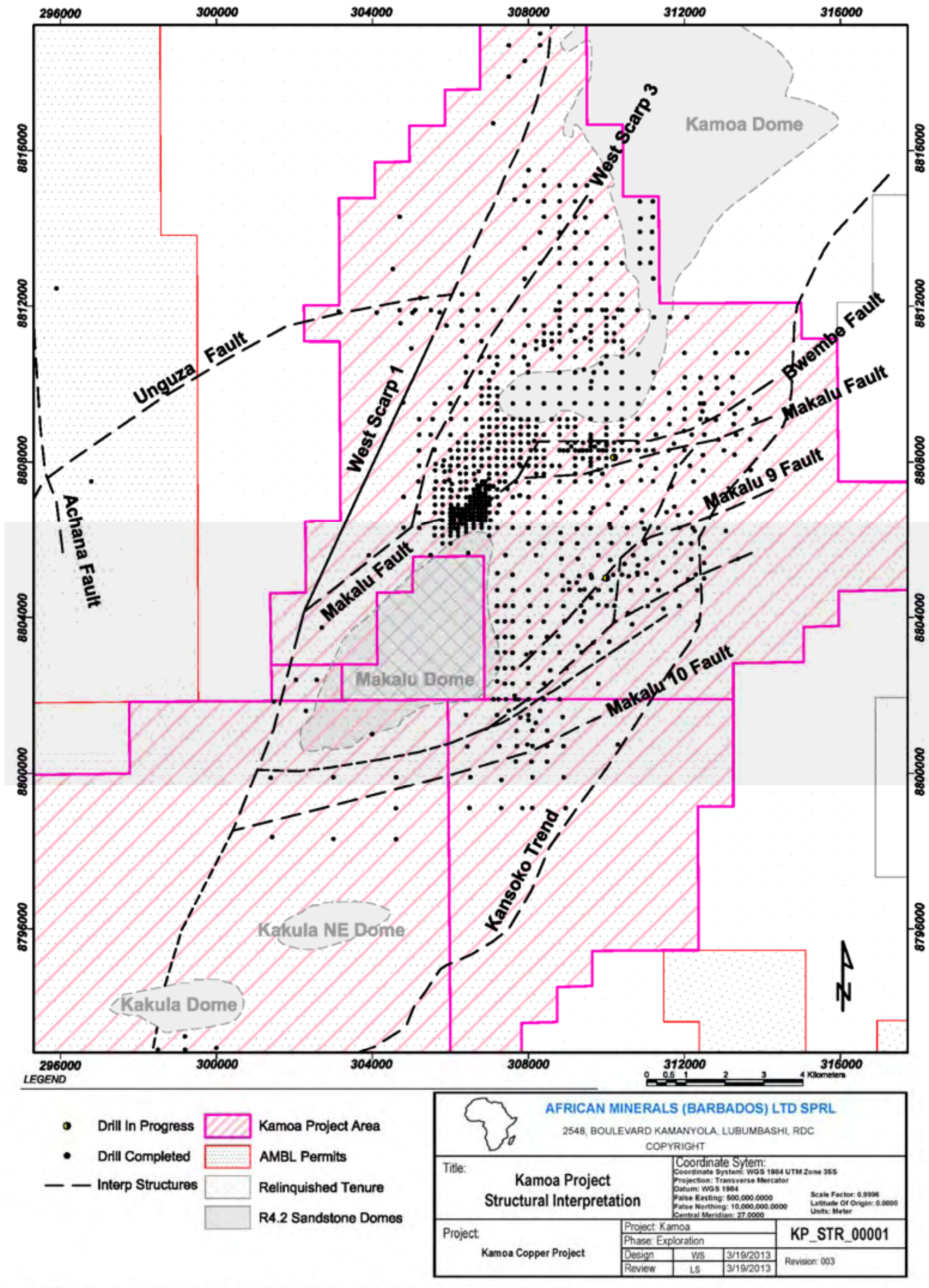
The base of the KPS (Ki1.1.2) has been used as a reference stratigraphic horizon during resource modelling. The centre of the mineralized intercepts used as the basis for the Mineral Resource estimate can occur above or below this reference horizon.

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: Drill hole 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.

Figure 7-3: Kamoa Project – Structural Interpretation (after Ivanplats)



Note: Figure courtesy Ivanplats, and redrawn by AMEC, 2013. The fault labelled as the West Scarp 1 Fault is commonly referred to simply as the West Scarp Fault.

Table 7-2: Stratigraphic Sequence for the Kamoa Project Area

Member	Unit	Lithology	Description	Thickness	Mineralization
Kalahari Sands			Superficial cover	up to 10 m	
Ki1.1: Grand Conglomerat	K1.1.3	Diamictite	Clasts usually range between 2 mm and 200 mm across, rarely in the 50–200 cm range, and are frequently dominated by quartzite, very fine-grained sandstone, shale and black argillite, very variable amounts of mafic rocks (mainly dolerite), with subordinate vesicular lava, quartz and scattered granitic and schistose basement cobbles. Coarse feldspathic sandstone clasts are rare and tend to be preferentially weathered. Rare clasts of Roan dolomite have also been recorded.	0 to >900 m	None
	K1.1.2	Pyritic siltstone-sandstone : Kamoa Pyritic Sandstone (KPS)	Predominantly well stratified and laminated, with 5% to >25% bedded pyrite. The basal contact with diamictite is frequently characterized by a thin (<20 cm) clast-supported quartzitic conglomerate. The lower portion always includes mudstone beds with abundant shale pellets.	15 m to 45 m	Occasionally near base
	K1.1.1	Ki 1.1.1.3	Clast-poor, silty/muddy and weakly carbonaceous diamictite that generally is well mineralized. Thicknesses vary across the Kamoa area with greater thicknesses seen in the South west. This unit contains good reductants which allows good precipitation of copper. As the unit is distributed over most of the Kamoa area it is key to the resource.	0 to 15 m	Well mineralized
		Ki 1.1.1.2	Sandstone–siltstone, thinly bedded, and only locally present.	0 to 5 m	Well mineralized
		Ki 1.1.1.1	Basal clast-rich, sandy diamictite, commonly oxidized.	0 to 30 m	Weakly mineralized
R4.2: Feldspathic Sandstone (RFS)			Feldspathic sandstone and arkose, often gritty, massive, and pebbly with some diffuse, relatively fine-grained conglomeratic bands (≤1m thick). Less than 5% of the unit consists of very fine-grained sandstone and thin hematitic silty bands; irregularly-bedded and rarely cross-bedded laminae. Significant proportions of the RFS are massive, and very coarsely bedded. Undulating and anastomosing laminae and finer silty zones occur irregularly. Normally well cemented, but in the top 1–5 m below the mineralized diamictite, it is often porous with obvious weathering of feldspar and dissolution of probable dolomite, accompanied by kaolinization, mostly due to secondary, geologically recent, processes.	0 to >200 m	None
R4.1: Poudingue (Roan Grit and Conglomerate)			Rests unconformably on Kibaran (basement) quartzite. Includes thin to >10 m thick conglomerate bands within pebbly, angular grit and sandstone beds. The sediments are typically immature and poorly sorted except for some of the conglomerate bands that have a weak feldspathic cement.	>100 m	None

Saprolite and Regolith Units

Thicknesses of as much as 10 m of superficial Kalahari sands occur mostly west of the West Scarp Fault and in the southern part of the Project area.

7.3.2 Structure

A fence diagram, along 8811900 N, shows the relationship of the lithologies and mineralization thicknesses encountered in the drilling hung on the base of the KPS (Ki1.1.2). The location of the section line is indicated in Figure 7-4. The mineralization intercepts shown on Figure 7-5, Figure 7-6 and Figure 7-7 were developed using a 1% Cu cut-off over a minimum 3 m of down-hole length. Thicknesses shown represent vertical thicknesses; true thicknesses will be slightly less than those shown. The fence diagram shows that the mineralization is continuous, but the detailed stratigraphic position of intercepts used for resource modelling can vary from drill hole to drill hole.

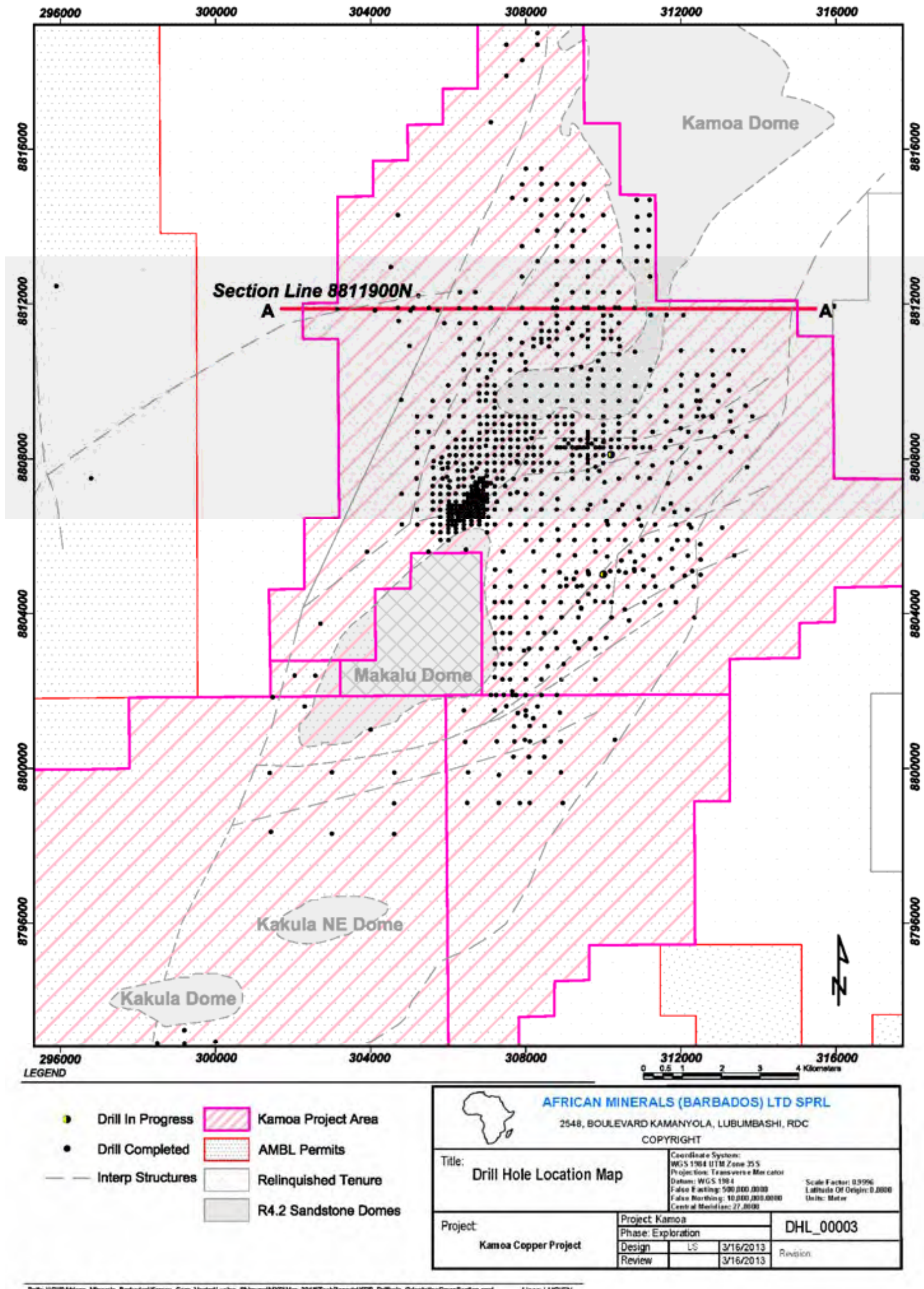
East of the Kamoa Dome along the Kansoko Trend, the Lower Kundelungu succession dips 15° to 21° to the south-east. To the west of the Kamoa Dome, the Ki1.1 strata dip at 5°to 9°, towards a basin centred close to drill hole DKMC_DD021 that is informally termed the Kamoa north-west basin.

The West Scarp Fault has a west-side down-throw of approximately 350 m to 400 m. West of the West Scarp Fault, the Ki1.1 strata appear to dip at less than 10° westward and southward. The West Scarp Fault is probably not a single plane, but a broad zone of brittle dislocations.

A third major magnetic structure, the Unguza Structure, forms a curvilinear shape, commencing east of drill hole DKMC_DD013 on line 8811900mN for approximately 11.5 km to the south-west as far as the Achana Fault. The structure has been identified by a prominent contrast in magnetic lineaments as well as a similar escarpment to that created by the West Scarp Fault.

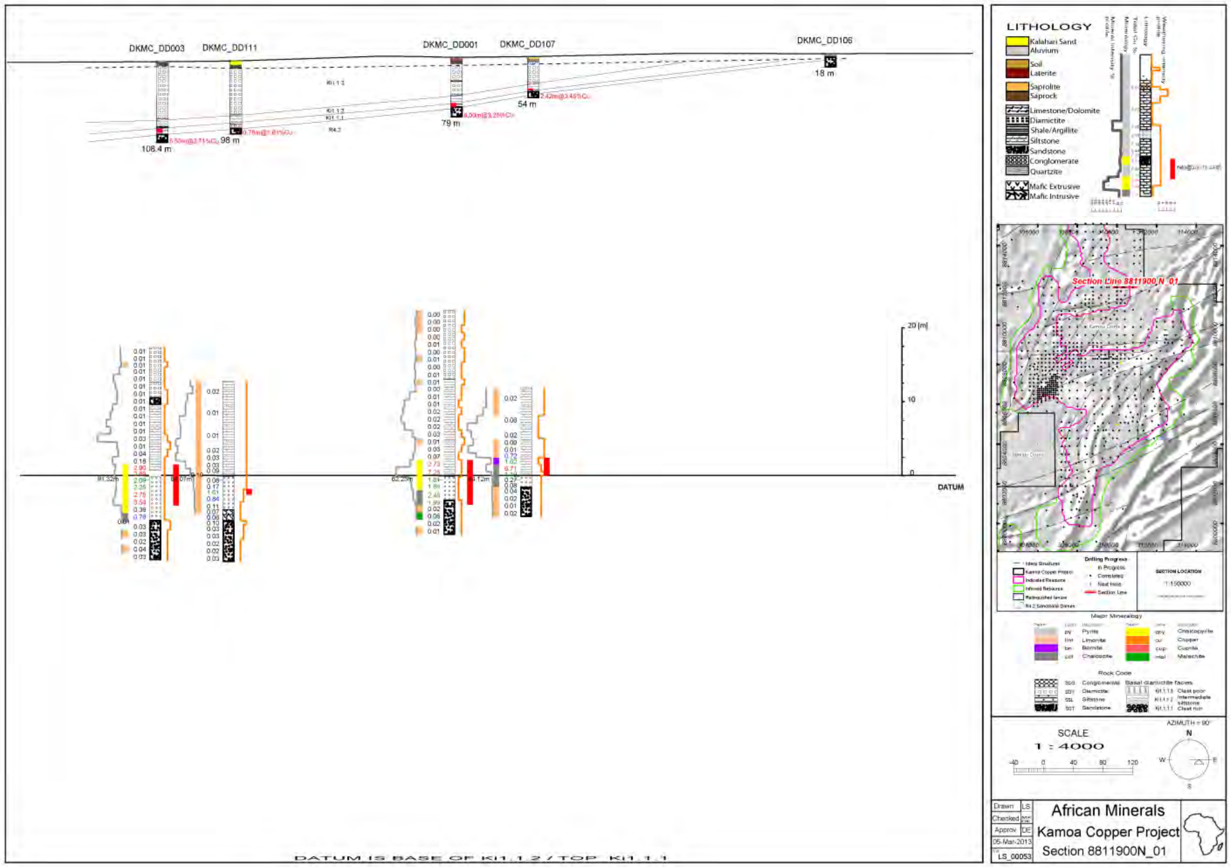
In the Mineral Resource area, approximately 25 structures have been identified by SRK based on geophysical data, and lithological discontinuities interpreted from the drill hole data. For the Mineral Resource estimate, these structures were spatially compared to the inflections in the geometry of the SMZ, and nine were identified as offsetting the mineralization. These structures were then used as boundaries to divide the mineralization into structural zones wherein the mineralization inside each structural zone maintains a similar strike and dip.

Figure 7-4: Drill Hole Location Map showing Orientation of Cross-Section and Drill Fences (8811900 N)



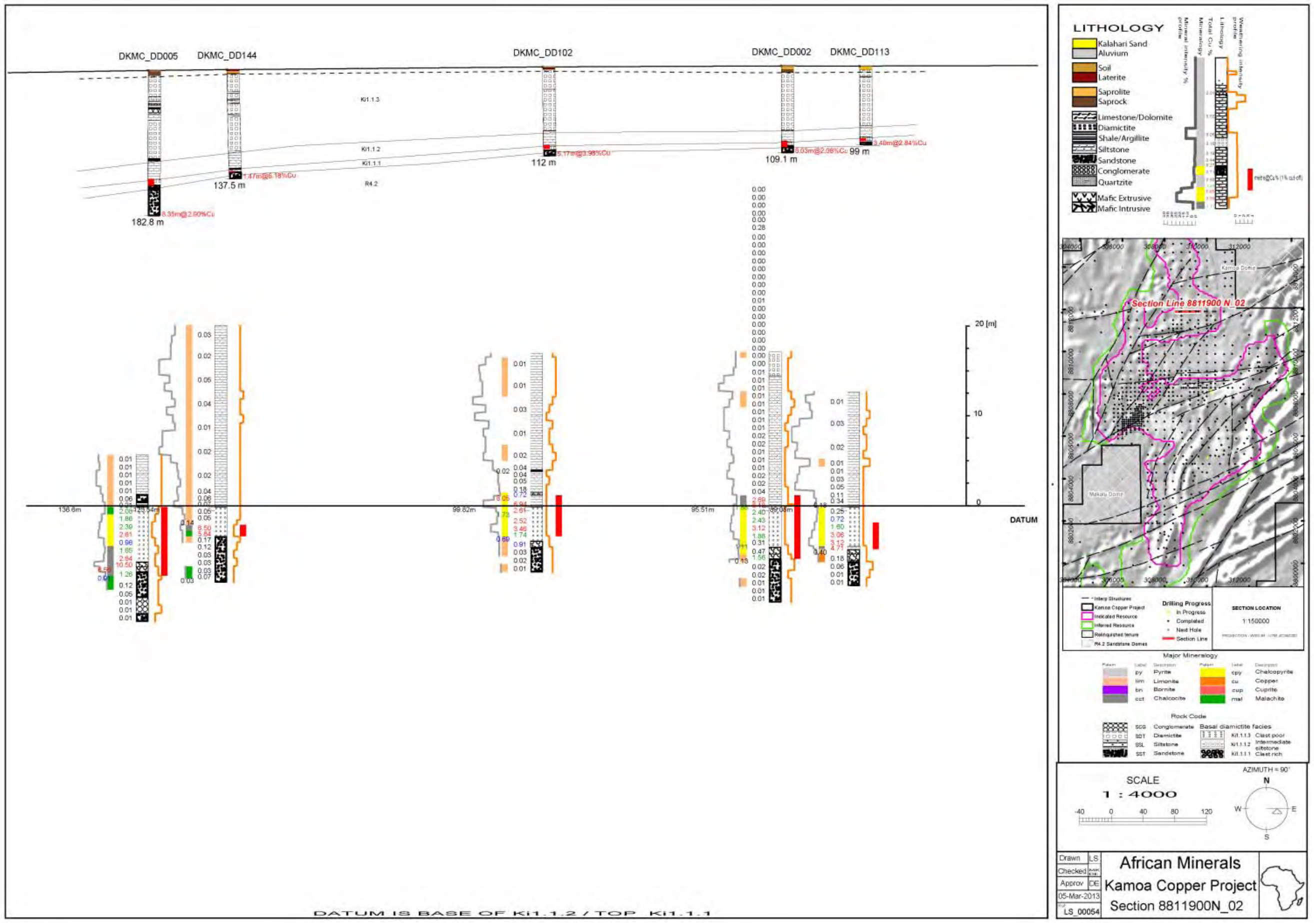
Note: Figure courtesy Ivanplats, modified by AMEC, 2013.

Figure 7-5: Drill Section at 8811900 N – Eastern Portion (section segment looks north)



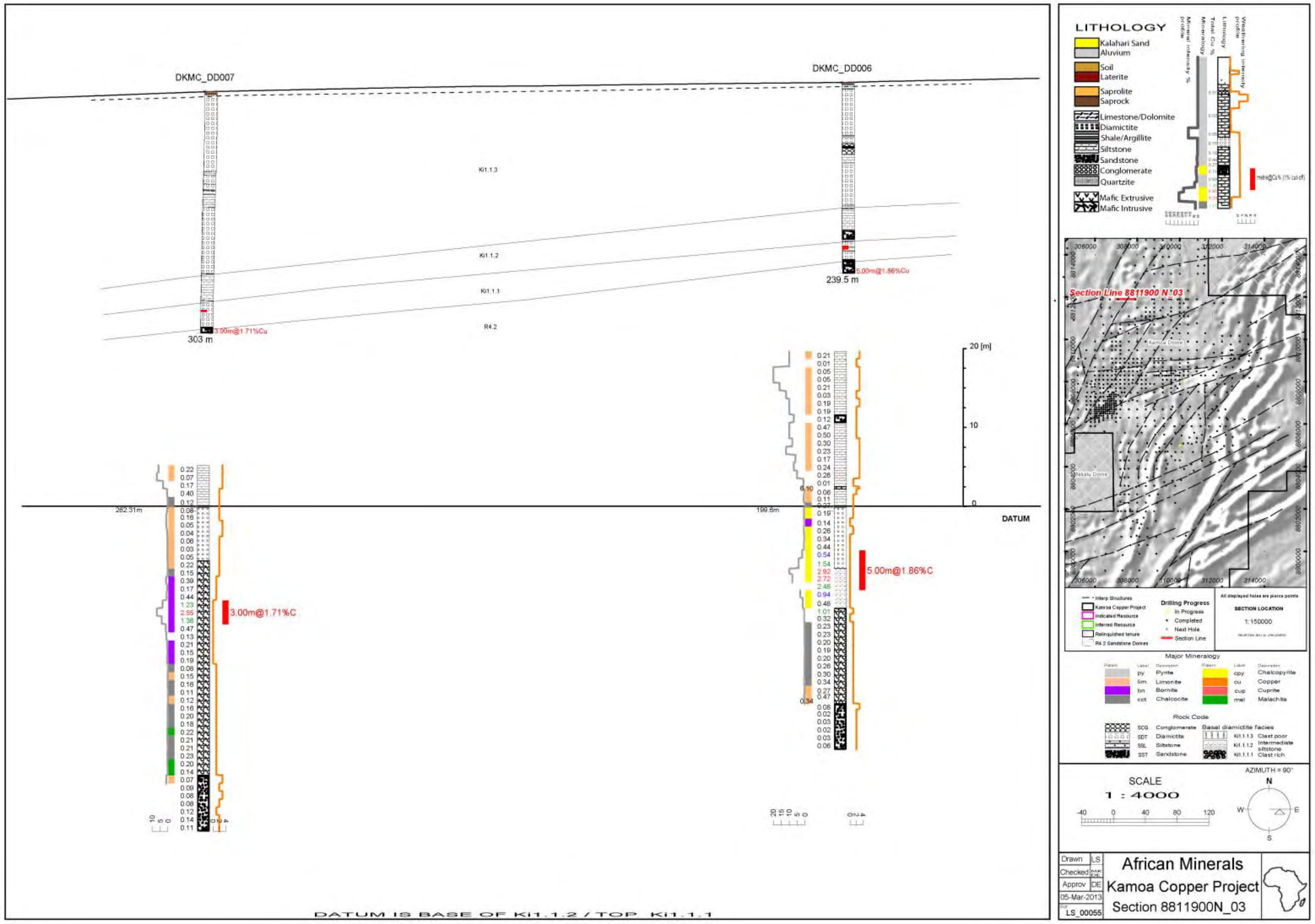
Note: Figure courtesy Ivanplats, modified by AMEC, 2013

Figure 7-6: Drill Section at 8811900 (Central Portion – section segment looks north)



Note: Figure courtesy Ivanplats, modified by AMEC, 2013

Figure 7-7: Drill Section at 8811900 N (Western Portion – section segments looking north)



Note: Figure courtesy Ivanplats, modified by AMEC, 2013

7.3.3 Metamorphism

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

7.3.4 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 Fault.

7.3.5 Mineralization

The genetic model developed by Ivanplats reflects modern interpretations for formation of the Copperbelt. During basin closure and broad folding, oxidizing saline brines migrated up-dip through Roan Supergroup rocks and leached copper. The brines encountered a redox boundary at the base of a diamictite, the Grand Conglomerat. This caused the precipitation of copper sulphide minerals in the basal Grand Conglomerat below a hanging wall interbed of pyritic siltstone–sandstone.

The mineralized sequence at the base of the diamictite comprises several interbedded units that appear to control mineralization (refer to Table 7-2).

From the base upward, the copper sulphides are zoned with chalcocite (Cu_2S), bornite (Cu_5FeS_4) and chalcopyrite ($CuFeS_2$). A lowermost clast-rich diamictite unit (Ki1.1.1.1) is thought by Ivanplats to be only weakly reducing, and generally hosts lower-grade (<0.5% Cu) mineralization throughout the unit. Copper sulphides tend to be of greatest abundance where this clast-rich diamictite is thin or absent, such that the overlying units lie directly on the Roan sandstones. The reducing capacity of the overlying clast-poor diamictite is considered by Ivanplats to be better than the lower clast-rich diamictite, and it is in contact with the reducing environment provided by the carbonaceous, pyritic diamictite above (Ki1.1.2). Most of the higher-grade mineralization occurs within the clast-poor unit (Ki1.1.1.3), or in the intervening, locally present, sandstone and siltstone (Ki1.1.1.2) that lies between the clast-poor and clast-rich units.

Near the surface, the diamictites have been leached. There may be some oxide copper and secondary sulphide enrichment down-dip, as for example in holes drilled adjacent to the edges of the Kamoa Dome. There are several kilometres of surficial boundary that present a target for this type of mineralization, which has been encountered to depths of 400 m in drill holes near faults. Some of the better-mineralized drill intercepts encountered to date are indicative of supergene enrichment. However, analogous deposits in Zambia such as Mufulira East, Nkana, and Chibuluma South had relatively narrow zones of oxides/supergene enrichment near-surface.

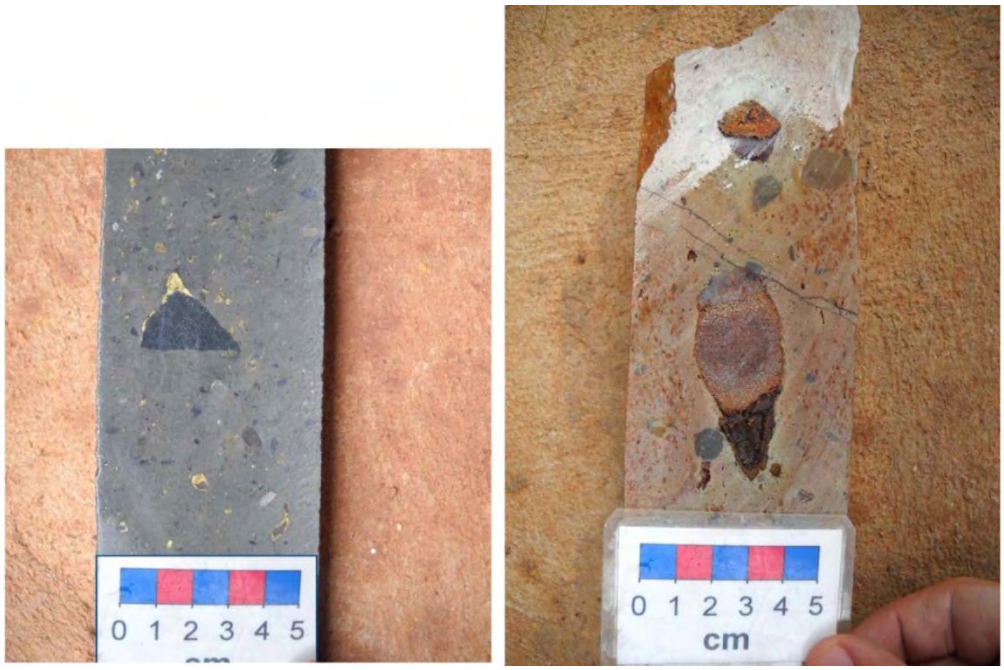
Hypogene Mineralization

Sulphides present at Kamoa are typical of sediment-hosted copper deposits, and display a characteristic vertical zoning away from the redox boundary (contact with R4.2 sandstones). Chalcocite, bornite, chalcopyrite, and pyrite share common bedding and replacement textures, ranging from semi-massive accumulations, to disseminations, to very coarse blebs, and occasionally can form veins. Sulphides commonly mantle and or partially replace clasts in the diamictite.

The three copper sulphide species and pyrite generally form overlapping zones, such that the lower, more copper-rich sulphide commonly mantles or partially replaces the overlying, more copper-poor sulphide (e.g. bornite mantles chalcopyrite in the overlap between these two mineral zones), particularly on the up-section side of clasts, where there may have been more permeability (low-pressure shadow). An example is shown in Figure 7-8.

Chalcopyrite, bornite and chalcocite are commonly concentrated in sandy lenses and planar beds within the Ki1.1.1.2 siltstone–sandstone interbed of the basal diamictite. This appears to reflect a local permeability control on the distribution of mineralization, and is also common in Zambian Ore Shale-hosted deposits. The footwall R4.2 (RFS) sandstone does not generally host significant hypogene mineralization, as it is located below the redox boundary.

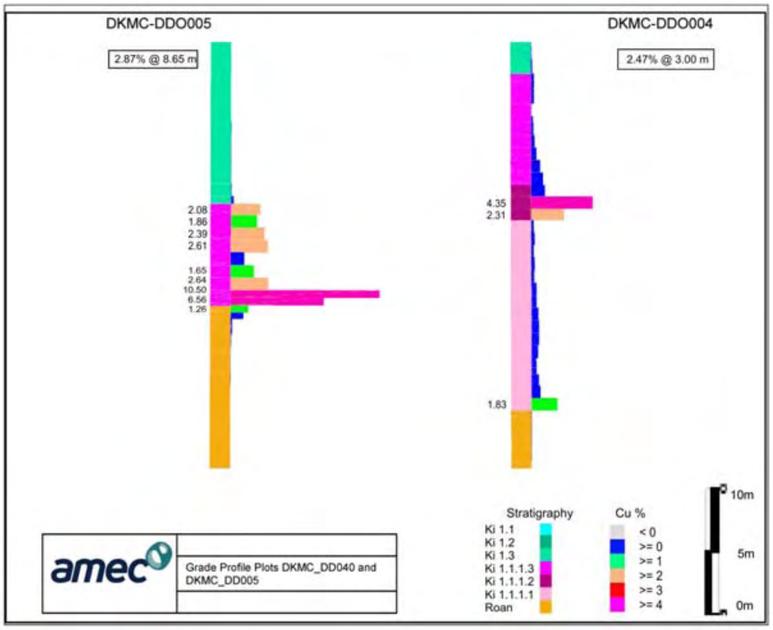
Figure 7-8 Diamictite Showing Clast Rimmed by Chalcopyrite (on left) and Oxide Equivalent (on right)



In general, the lowermost Ki1.1.1.1 clast-rich sandy diamictite typically hosts weakly disseminated copper mineralization, grading <0.5% Cu; whereas the Ki1.1.1.3 silty clast- poor diamictite hosts better copper grades. Vertical sulphide zoning is developed regardless of this facies control. Hence, where the basal clast-rich Ki1.1.1.1 subunit is thicker, it is common to observe low-grade chalcocite and bornite mineralization in the basal Ki1.1.1.1 diamictite, and high-grade chalcopyrite mineralization in the overlying Ki1.1.1.2 siltstone–sandstone interbed and Ki1.1.1.3 diamictite.

Figure 7-9 compares grade profiles for two drill holes, DKMC-DD004 and DKMC DD005.

Figure 7-9: Grade Profiles for DKMC-DD004 and DKMC DD005



In drill hole DKMC-DD004, there is a thick sequence of clast-rich diamictite (Ki1.1.1.1); the copper grades are diffuse, and there is a sharp increase in grade going into the siltstone-sandstone interbed Ki1.1.1.2, but only over 2 m. The copper-bearing solutions mainly precipitated copper sulphides in the less reduced, less permeable, clast-poor diamictite.

In contrast, drill hole DKMC-DD005 has no clast-rich diamictite, and copper-bearing solutions precipitated a rich intercept of copper sulphides within the more reduced and permeable clast-poor diamictite. Proximity to pyritic, carbonaceous material within the KPS (Ki1.1.2) unit with pyritic, carbonaceous material probably contributed to formation of a very strong redox boundary.

For most drill holes at Kamoa, the grade profile shows a sharp increase from less than 0.5% Cu to over 1% Cu at the top and bottom of continuous zones grading over 1% Cu. Thus 1% Cu is a “natural” cut-off, and this is also typical of deposits of the Zambian Copperbelt and the shale-hosted copper deposits of northern Michigan, USA.

Virtually all of the significant copper mineralization occurs within the basal Ki1.1.1 diamictite sequence, with the vertically-overlying pyrite zone occurring within the Ki1.1.2 siltstone–sandstone (KPS). This unit contains 5% to >25% pyrite, both as obvious, very coarsely recrystallized, subhedral to euhedral grains, and as very fine-grained, possibly framboidal disseminations. The lowermost 3 m to 5 m of the Ki1.1.2

unit generally contains only fine-grained pyrite, but it may also locally host copper mineralization.

Carrollite ($Cu(Co,Ni)_2S_4$) is common in the Mines Subgroup deposits of the Katangan Copperbelt, and in many of the Zambian Ore Shale deposits, but to date is rare at Kamoa.

Sphalerite ($(Zn,Fe)S$) mineralization is common, but weakly developed (typically<0.1% Zn), in the transition between the chalcopyrite and overlying pyrite zones. Sphalerite displays the same textures as the copper sulphides, indicating it is part of the same mineralizing process. West of the West Scarp Fault, sphalerite can form distinct and higher-grade zones in siltstone–sandstone interbeds within the diamictite.

Supergene and Secondary Mineralization

In many instances chalcocite occurs in the same manner as bornite and chalcopyrite, and is considered a hypogene (primary) phase. However, in shallow intersections or in areas of deep weathering, chalcocite also forms distinct zones, unrelated to the normal vertical hypogene zoning, and is interpreted as a supergene phase. In such cases it can be recognized by association with other supergene phases, such as cuprite and native copper, and by its mantling iron-bearing sulphides (chalcopyrite and pyrite) in contrast to the normal paragenetic sequence.

Cuprite (Cu_2O) is uncommon and normally occurs in the same environment and mode as supergene chalcocite, but as tiny specks, as blebs in contact with chalcocite, and as fracture fillings and along rare bedding planes. It most commonly occurs in the basal zone just above the RFS (R4.2) and is a hallmark of secondary copper redistribution. Native copper is found as crystalline plates along oxidized joints and occasionally as specks and veinlets, particularly within a few decimetres of the RFS. It commonly replaces chalcocite in this horizon and has been found as bedding-parallel veinlets in siltstone and replacement blebs in weathered basal diamictite. Native copper seldom constitutes >1.0% of the total volume of any zone.

Malachite ($Cu_2(CO_3)(OH)_2$) is uncommon, occurring mostly in fractures related to faults and joints, and replacing oxidized copper sulphides in weathered clasts. It also forms by weathering of chalcocite, and has been redistributed into the RFS in a few places along joints and in the porous matrix, accounting for 0.1–0.4% Cu.

Azurite ($Cu_3(CO_3)_3(OH)_2$), chrysocolla ($(Cu,Al_2)H_2Si_2O_5(OH)_4 - nH_2O$), and libethinite ($Cu_2(PO_4)(OH)$) have also been identified.

7.4 Comments on Section 7

The AMEC QPs note the following:

- The understanding of the deposit setting, lithologies, and geological, structural, and alteration controls on mineralization is sufficient to support estimation of Mineral Resources.
- Mineralization at Kamoa has been defined over an irregularly-shaped area of 20 km x 15 km. The mineralization is typically stratiform, and vertically zoned. 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. The stratigraphic position of the mineralized intercepts vary on a local basis.
- Geological controls on the mineralization, the mineralization style, mineralization setting, and the mineralogy are sufficiently well understood to support declaration of Mineral Resources
- Typically contaminants are not a problem for Copperbelt deposits. The initial 2010 drilling program 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 mineralization occurs near the contact with the KPS. This assaying was discontinued by Ivanplats in 2010–2011, because AMEC considered contaminant levels to be low. Reid (2010a) showed good correlation between trace and minor element assays with Niton (X-ray fluorescence or XRF) results. Therefore, Niton results for holes drilled since 2010 should be adequate to identify any areas where contaminants would be of concern.

8.0 DEPOSIT TYPES

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

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

- Geological setting: Intracratonic rift; fault-bounded graben/trough, or basin margin, or epicontinental shallow-marine basin near paleo-equator; partly evaporitic on the flanks of basement highs; sabkha terrains; basal sediments highly permeable. Sediment-hosted stratiform copper deposits occur in rocks ranging in age from Early Proterozoic to late Tertiary, but predominate in late Mesoproterozoic to late Neoproterozoic and late Palaeozoic rocks
- Deposit types:
  - Kupferschiefer-type: host rocks are reduced facies and may include siltstone, shale, sandstone, and dolomite; these rocks typically overlie oxidized sequences of hematite-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 (hematite-bearing) and grey (relatively reduced, commonly pyrite-bearing) sandstone, arkose, or conglomerate. The configuration of the mineralized zone varies from sheet-like, with extensive horizontal dimensions, to tabular or roll-front geometries, with limited horizontal dimensions
- Mineralization: Deposits consist of relatively thin (generally <30 m and commonly less than 3 m) sulphide-bearing zones, typically consisting of hematite–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

- Mineralization timing: Sulphides and associated non-sulphide minerals of the host rocks in all deposits display textures and fabrics indicating that all were precipitated after host-rock deposition. Timing of mineralization relative to the timing of host-rock deposition is variable, and may take place relatively early in the diagenetic history of the host sediments or may range to very late in the diagenetic or 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
- Mineralization controls: Reducing low pH environment such as marine black shale; fossil wood, and algal mats are important as well as abundant biogenic sulphides and pyritic sediments. High permeability of footwall sediments is critical. Boundaries between hydrocarbon fluids or other reduced fluids and oxidized fluids in permeable sediments are common sites of deposition
- Alteration: Metamorphosed red-beds may have a purple or violet colour caused by finely disseminated hematite.

8.1 Comments on Section 8

Many features of the mineralization identified within the Project to date are analogous with the Polish Kupferschiefer-type deposits and the 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 Fault and Kansoko Trend
- Strong host-rock control and restriction of the mineralization to a redox boundary zone between oxidized footwall hematitic sandstone and reduced, sulphidic host diamictites and siltstone-sandstone rocks

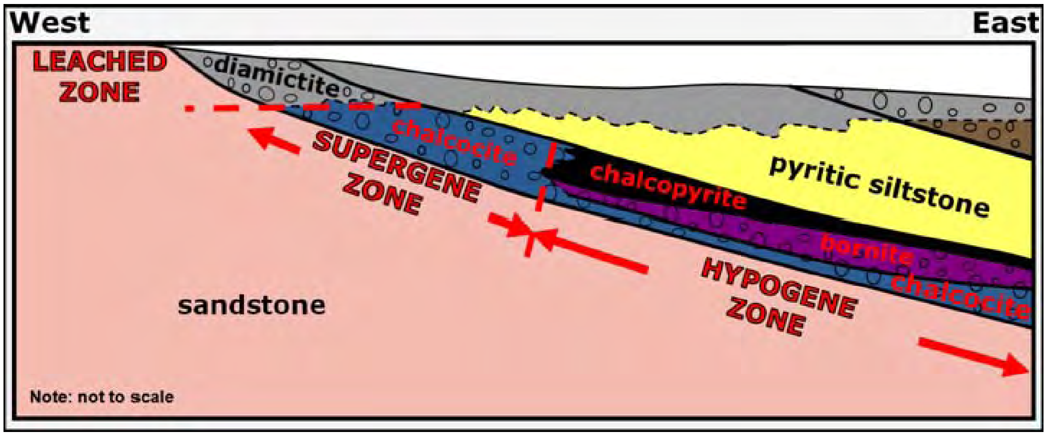
- Presence of the replacement, blebby, and matrix textures that are typical of sediment-hosted copper deposits
- Vertical zoning of disseminated copper sulphide minerals from chalcocite to bornite to chalcopyrite; association with cobalt, silver (but thus far not in economically significant quantities); refer to Figure 8-1
- Hypogene minerals are chalcopyrite, bornite and chalcocite, with the predominant copper sulphide species varying spatially throughout the deposit. For example recent 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 mineralization.

The virtual absence of carbonate rocks and the absence of widespread silicification both as host-rock alteration and in veins is atypical of the Mines Subgroup-hosted deposits of the Katangan Copperbelt (e.g. Tenke–Fungurume). Minor dolomite replacement of sulphidic clast rims in the basal diamictite and scattered tiny carbonate ± quartz veinlets with occasional sulphides shows that this process mineralization.

The Kamoa deposit is currently unique within the DRC 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.

The AMEC QPs consider the deposit model developed by Ivanplats for the Project is appropriate to the style of mineralization that has been identified.

Figure 8-1: Mineralization Zonation Schematic, Kamoa Deposit



Note: Figure is schematic and not to scale. The true thicknesses of drilled composites ranges 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 courtesy Ivanplats, 2012.

9.0 EXPLORATION

Exploration work on the Project was conducted by Kamoa Project staff, under the supervision of African Mining Consultants, an independent consulting firm based in Kitwe, Zambia.

Prior to commencement of on-ground exploration in 2004, Ivanplats 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 Ivanplats' 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. Ivanplats obtained higher resolution, 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 Ivanplats to be a basal conglomerate of the Lower Roan Poudingue, located near its unconformable contact with Kibaran quartzite below.

A reconnaissance field mapping program occurred between August and October, 2010 at the Kakula Prospect, situated south of the Makalu Dome. The purpose of this program 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 programs have delineated elevated copper associated with this contact. Mapping has successfully delineated the contact, and this information will be used for planning future drilling for the area.

9.3 Geochemical Sampling

Geochemical and aircore drill sampling programs were conducted as part of first-pass exploration and used to help vector into mineralization. Geochemical sampling programs 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 Ivanplats. The data recorded included total field magnetic intensity, horizontal and longitudinal magnetic gradient, multi-channel radiometric, linear and barometric, altimetric and positional data. The survey was flown at a terrain clearance of 80 m, and covered an area of 7,900 km^2, for approximately 36,775 line kilometres of survey. The Project area is a small portion of the survey area. Tie lines were spaced at 2,500 m, and the tie line trend was 225°. The traverse lines were spaced at 250 m, and the traverse line trend was 135°. Quality control of the data was maintained during both survey programs by independent consulting geophysicist Steven McMullan. Data processing was completed using Oasis Montaj software from Geosoft Inc. of Toronto, Canada.

In 2011, Gap Geophysics Australia and Quik_Log Geophysics conducted down hole 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 down hole 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 in Kamoa 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 drill holes. Results were inconclusive.

A ground magnetic survey was completed over the Kamoa area during 2011–2012. Instruments purchased by AMBL were used by locally-employed teams. The ground magnetic data are being 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 Ivanplats in 2009 from the mineralized zone and footwall sandstone in drill hole DKMC_DD019. Analyses were completed at Ultra Trace laboratories, and included a standard (10 element plus SO_3 and LOI) X-ray fluorescence (XRF) major element suite, and 46 element inductively coupled plasma (ICP) trace element suite. Results indicated possible K_2O enrichment commensurate with potassic (feldspar–sericite) alteration.

An MSc study focused on the petrology and microscopy of the mineralized zone and the pyritic hanging wall is underway at the Colorado School of Mines. A secondary topic of study will be the sedimentology and stratigraphy of the diamictites.

9.6 Geotechnical and Hydrological Studies

Geotechnical studies completed in support of the 2012 PEA are discussed in Section 16.

9.7 Metallurgical Studies

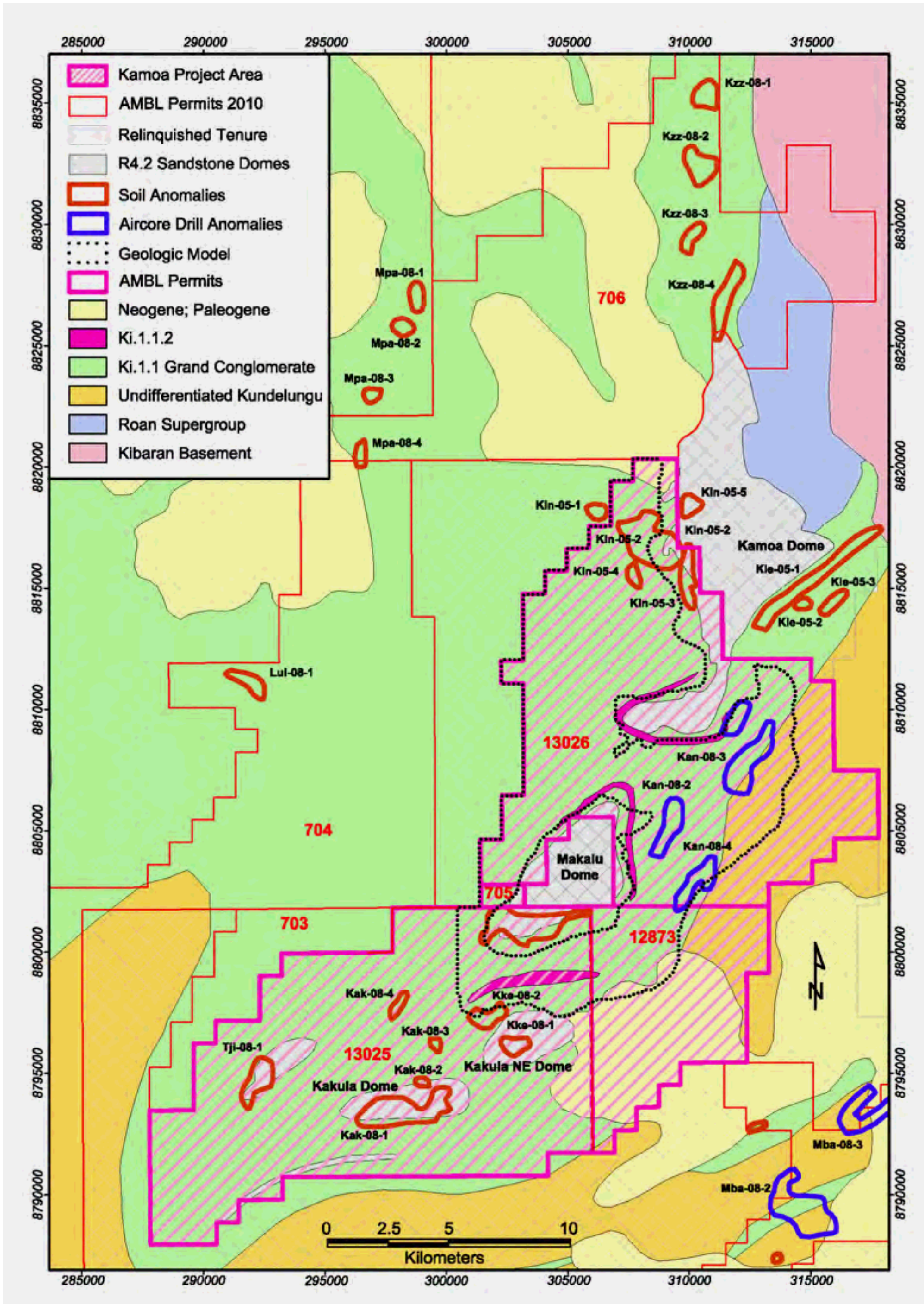
Metallurgical studies and testwork completed on the Project are summarized in Section 13.

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 mineralization, and for new zones of mineralization within this same horizon.

Exploration identified a number of priority grass-roots exploration prospects within the Project (Figure 9-1), based on geological and geophysical 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 along strike to the southwest, and are analogues of the Kamoa and Makalu Domes where the Kamoa mineralization was initially discovered.

Figure 9-1: Location Map, Geochemical and Aircore Drill Anomalies



Note: Figure courtesy Ivanplats, modified by AMEC, 2013.

To test its exploration concept, Ivanplats conducted regional widely-spaced exploration drilling in 2009 in the area underlain by diamictite west and north of the 2009 Kamoa Inferred Mineral Resource. While copper mineralization was commonly intersected in the basal part of the diamictite, grades and thicknesses were generally significantly lower than were found at Kamoa.

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 mineralization below the diamictite hanging wall, Ivanplats considers that the most reliable means of evaluating this hypothesis is with wide-spaced drill holes. This approach is predicated on the assumption that stratiform copper deposits are laterally extensive, and are at the kilometre scale.

9.9 Comments on Section 9

In the opinion of the AMEC QPs:

- The exploration programs completed to date are appropriate to the style of the Kamoa deposit
- The provisional research work that has been undertaken supports Ivanplats' genetic and affinity interpretations for the Project area
- The Project area remains prospective for additional discoveries of base-metal mineralization around known dome complexes
- Anomalies generated by geochemical, geophysical and drill programs to date support additional work on the Project area.

10.0 DRILLING

Drilling on the Project has been undertaken in a number of aircore, RC, and core campaigns from May 2006. Aircore and RC holes were used for reconnaissance follow-up of geochemical anomalies. Core holes were used for reconnaissance exploration and deposit delineation, and for Mineral Resource estimation. Drill programs were completed by contract drill crews, typically supervised by African Mining Consultants or Ivanplats staff.

The drill database used for Mineral Resource estimation was closed at 10 December 2012. Drilling has been ongoing since that date, and information on the program to 11 March 2013 is included in this Report. Drill hole collar locations are shown for the entire Project in Figure 10-1. Drilling is summarized in Table 10-1. For the remainder of this report, the 15 RC pre-collar (RD) holes are included with the core drill holes in any reference to core drilling. Two core holes were being drilled as at 11 March 2013.

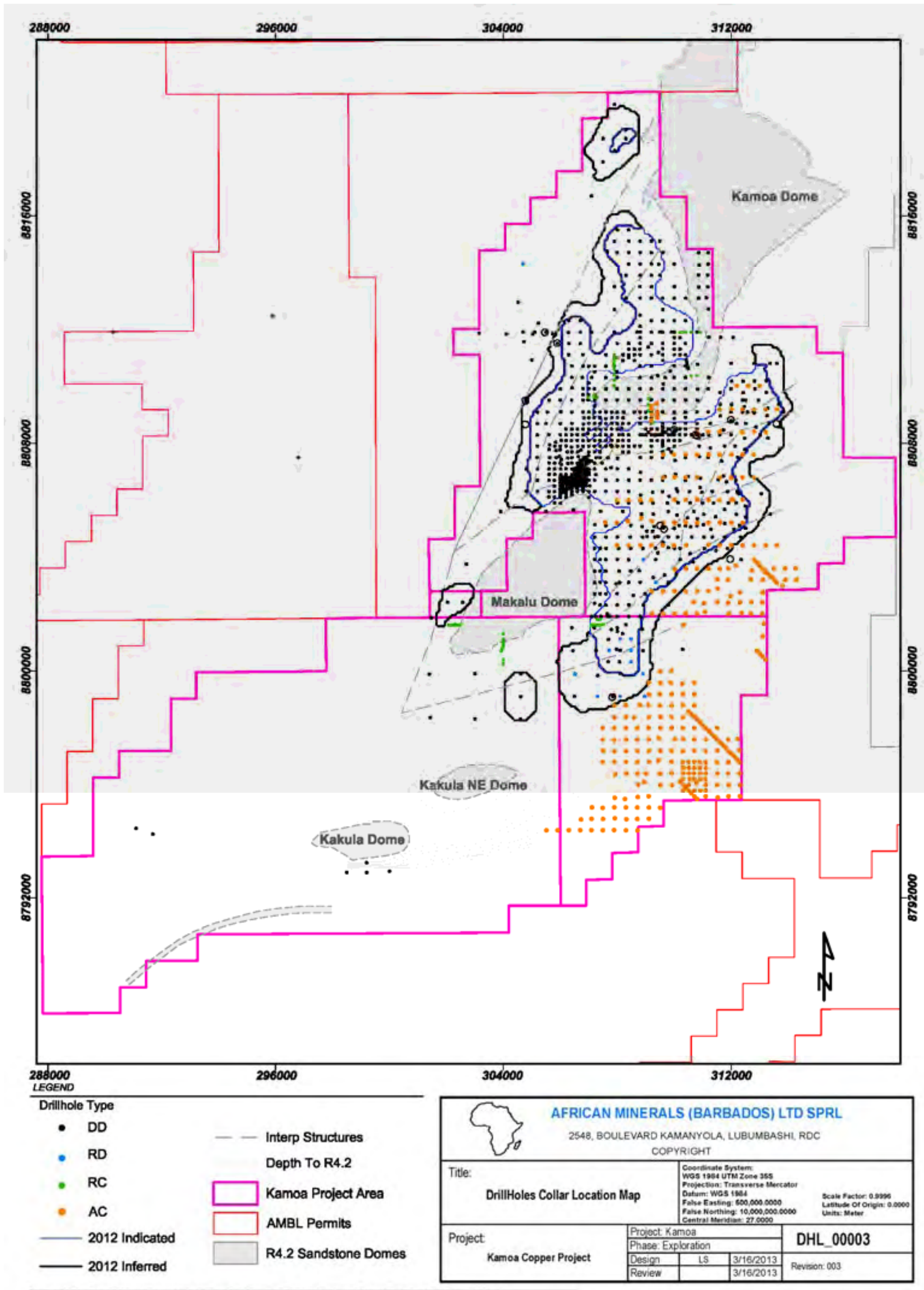
As at 11 March 2013, there were 842 core holes within the Project area boundaries (including five close-spaced wedged deflections and six twin holes), Figure 10-2. Of these drill holes, a total of 555 are used to support the December 2012 Mineral Resource estimate (543 drill holes are in Domain1; area where Indicated and Inferred Mineral Resources are estimated) plus 12 are in Domain 2; the area where the Exploration Target is estimated). The 287 core holes that were not used to support the December 2012 Mineral Resource estimate were excluded because they were abandoned, unmineralized holes used to define the extents of the mineralization, not sampled (e.g. hydrogeology holes), or because assays were pending at the time of database closure.

At the time of the completion of the September 2011 Mineral Resource estimate on which the 2012 PEA is based, a total of 720 holes lay within the boundaries of the then-defined Project; 309 core holes (including five close-spaced wedged deflections and six twin holes) are spaced close enough together (spacing less than approximately 800 m) to allow Mineral Resource estimation (Figure 10-3). A total of 330 core holes were used to model the stratigraphy and Exploration Targets within the Kamoa Project for the September 2011 Mineral Resource estimate.

10.1 Drill Methods

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).

Figure 10-1: Drill Hole Collar Location Map Kamoa Deposit Area

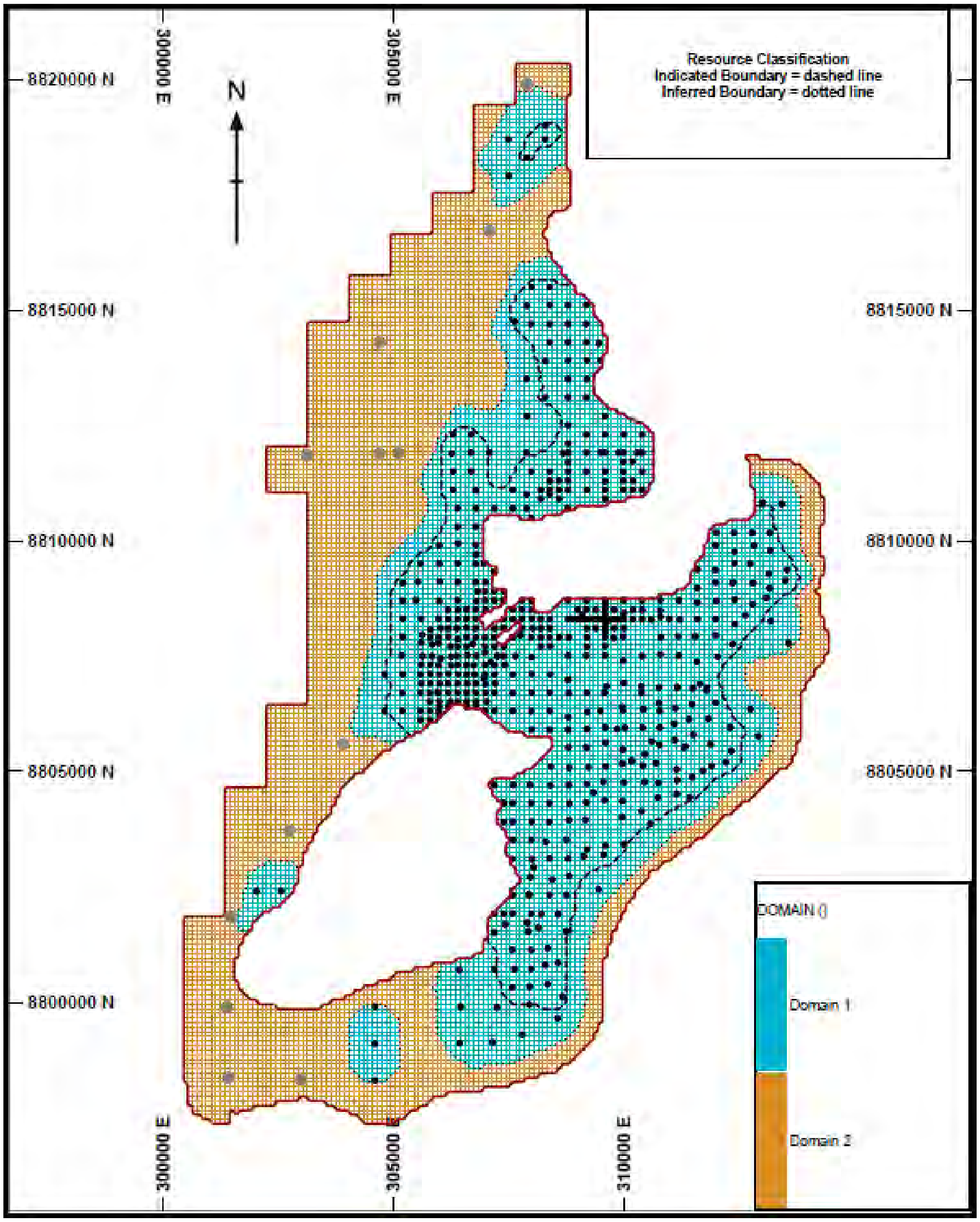


Note: Figure courtesy Ivanplats, modified by AMEC, 2013.

Table 10-1: Drilling Summary Table (March 11, 2013)

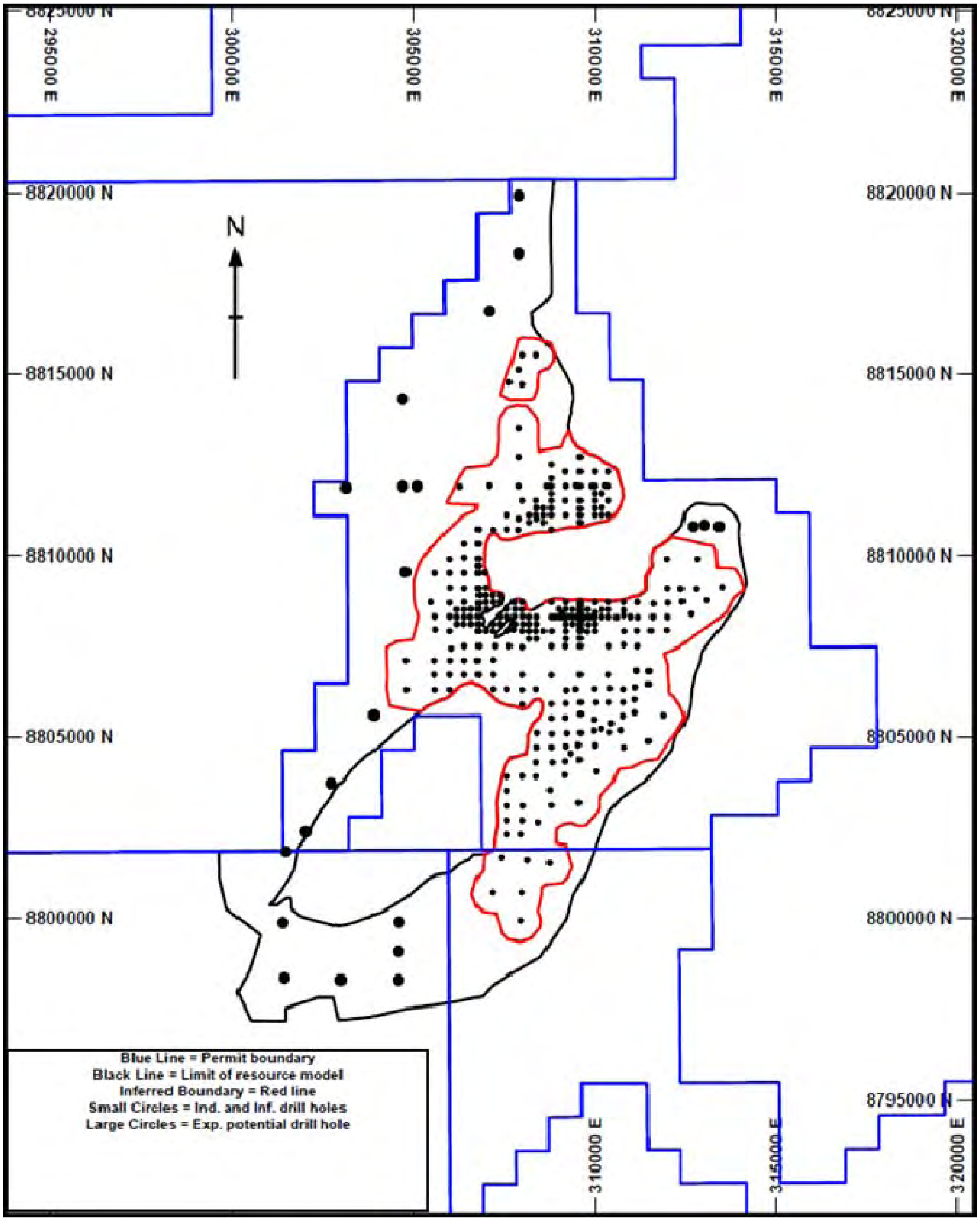
Permit	Aircore	Metres	Core	Metres	Reverse Circulation	Metres	Reverse Circulation Collar, Core Tail	Metres	Reverse Circulation Water bore	Metres	Rotary Air Blast	Metres
PE 12873	176	4585.0	23	8211.1	7	867.0	10	6930.5	0	0.0	0	0.0
PE 13025	2	49.0	17	6539.0	13	1358.0	0	0.0	0	0.0	0	0.0
PE 13026	138	2776.0	785	184398.2	25	2479.0	5	1889.0	2	208.0	47	8933.7
PR 705	0	0.0	2	331.5	0	0.0	0	0.0	0	0.0	0	0.0
Total	*316*	*7,410.0*	*827*	*199,479.8*	*45*	*4,704.0*	*15*	*8,819.5*	*2*	*208.0*	*47*	*8,933.7*

Figure 10-2: Collar Location Map showing Drill Holes used in December 2012 Mineral Resource Estimate



Note: Figure prepared by AMEC, 2013. Small circles are drill holes that lie within Domain 1 and large circles are drill holes that lie within Domain 2. Grid is approximately 5 km x 5 km

Figure 10-3: Collar Location Map showing Drill Holes used in September 2011 Resource Model that Supports the 2012 PEA



Note: Figure prepared by AMEC, 2012

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, mineralization, structural and geotechnical information. Logged data are then entered into Excel spreadsheets using single data entry methods by African Mining Consultants personnel. A hand-held Niton XRF instrument has been used by African Mining Consultants during drill hole logging from 2007 onwards 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 will eliminate delays and errors associated with data entry of paper logging forms.

A new revised “Major Structure” logging scheme has been initiated based on recommendations by SRK Consultants. These data complement the logging form described above and are captured electronically.

Historic data from oriented core logging will be integrated with these data. Highlights of the “Major Structure” log are:

- Populates two databases, one based on intervals (From/To) and one based on point data (Depth and optional Thickness). The intervals table will mainly contain faults and other damaged zones, while the points table will contain veins, bedding, and discrete fractures/joints
- Data are captured for the entire drill hole. Where possible, orientated data will be captured

- Logging is based on the philosophy of "structural core mapping", attempting to create an interpreted structural log that illustrates the "character" of the core in a particular drill hole, with the main focus being fault and damage zones. For the interval table, the result is a brief summary log of structures for most holes, and this is complimented by the points table and by the "Detailed Geotech" log being produced by geologists
- Historic core from key areas within the project is being re-logged. If noted, new data are added to the KCP Geologic Logging Form.

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

10.3 Core Handling

Core was obtained using wire-line methods and was washed prior to placement in core trays. Aluminum core trays were placed near the core barrel so that the core was placed in the tray in the same orientation as it came out of the barrel. Rubble, which was rarely encountered, was piled to about the length of the whole core that its volume would represent. Trays were marked with the prospect/deposit name, drill hole identity number, the intersection interval (start and final depths in that box), an arrow indicating which side is down-the-hole, and the sequential box number (e.g. Box 6 of 57).

Any break in the core made during removal from the barrel was marked with an "x" on the core. When breakage of the core was required to fill the box, edged tools were used, and the end of every run was marked.

Transport of core boxes to the core shed was undertaken daily by Ivanplats personnel supervised by either Ivanplats' or African Mining Consultants' geologists. Core-handling logs were completed that included documentation of all personnel involved in any step during the logging and sampling procedures.

10.4 Recovery

Core recovery was determined prior to sampling. Typically, a recovery measurement was completed on-site prior to transporting the core, and before detailed logging was completed. Standard core recovery forms were usually completed for each hole by the field assistant or geologist. Core recovery was also measured by the driller and included in drilling records.

Core recovery in the mineralized units ranges from 0% to 100% and averages 95%. The 0% recovery is likely due to missing data, as logging does not indicate poor recovery. Visual inspection by AMEC documented the core recovery to be excellent.

10.5 Collar Surveys

All drill sites surveyed were supervised by either Ivanplats or African Mining Consultants personnel using a hand-held GPS that is typically accurate to within about 7 m.

From 2010 through 2012, all completed holes were surveyed by an independent professional surveyor, SD Geomatique, using a differential GPS that was accurate to within 20 mm. Since 11 December, 2012, Ivanplats has added an additional 87 holes to the database, but based on the co-ordinates, only three of these holes have been surveyed. It is expected that SD Geomatique will complete the collar surveys in due course.

10.6 Down-hole Surveys

Core hole orientations ranged from azimuths of 0° to 360°, with down-hole inclinations that ranged from -24.5º to vertical. The inclination of -24.5° was contained in hole DKMC_DD452 at a depth of 1,137 m. Most holes were vertical or subvertical, with 712 of the core holes having collar inclinations that ranged from -50° to vertical.

Core hole depths ranged from a minimum of 42 m to a maximum of 1,706 m, averaging about 250 m.

Down-hole surveys for most drill holes were performed by the drilling contractor at approximately 30 m intervals for 2009 drilling and at 50 m intervals for 2010 through 2013 drill holes, using a Single Shot digital down-hole instrument that is calibrated every two weeks. The instrument records inclination, azimuth, temperature and magnetic susceptibility at each survey depth. 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 down-hole surveyed. These holes were either short holes (total depth less than 100 m) or abandoned holes, and the missing surveys do not materially impact the Mineral Resource estimate.

The down-hole survey readings are recorded by the driller on the daily drilling record sheets, and the survey certificate supplied by the drilling contractor. The site geologist then enters the down-hole data on the drill hole summary sheet.

A total of 404 of the 555 core holes used in resource modeling (see Section 14) were vertical holes (inclination less than -80°) ranging in total depth from 42 m to 1,706 m, and have end-of-hole deviations averaging 3.4 m in easting and 0.4 m in northing. The maximum deviation was 117 m west noted in drill hole DKMC_DD577, which had a total depth of 1,439 m, and 63.4 m north, noted in drill hole DKMC_DD617, which had a total depth of 1,143 m. The remaining 151 holes used in resource modeling were inclined with inclinations ranging from -80° to -24.5°. The average deviation for the inclined holes was 0.7° per 100 m.

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

10.6.1 Oriented Drill Core

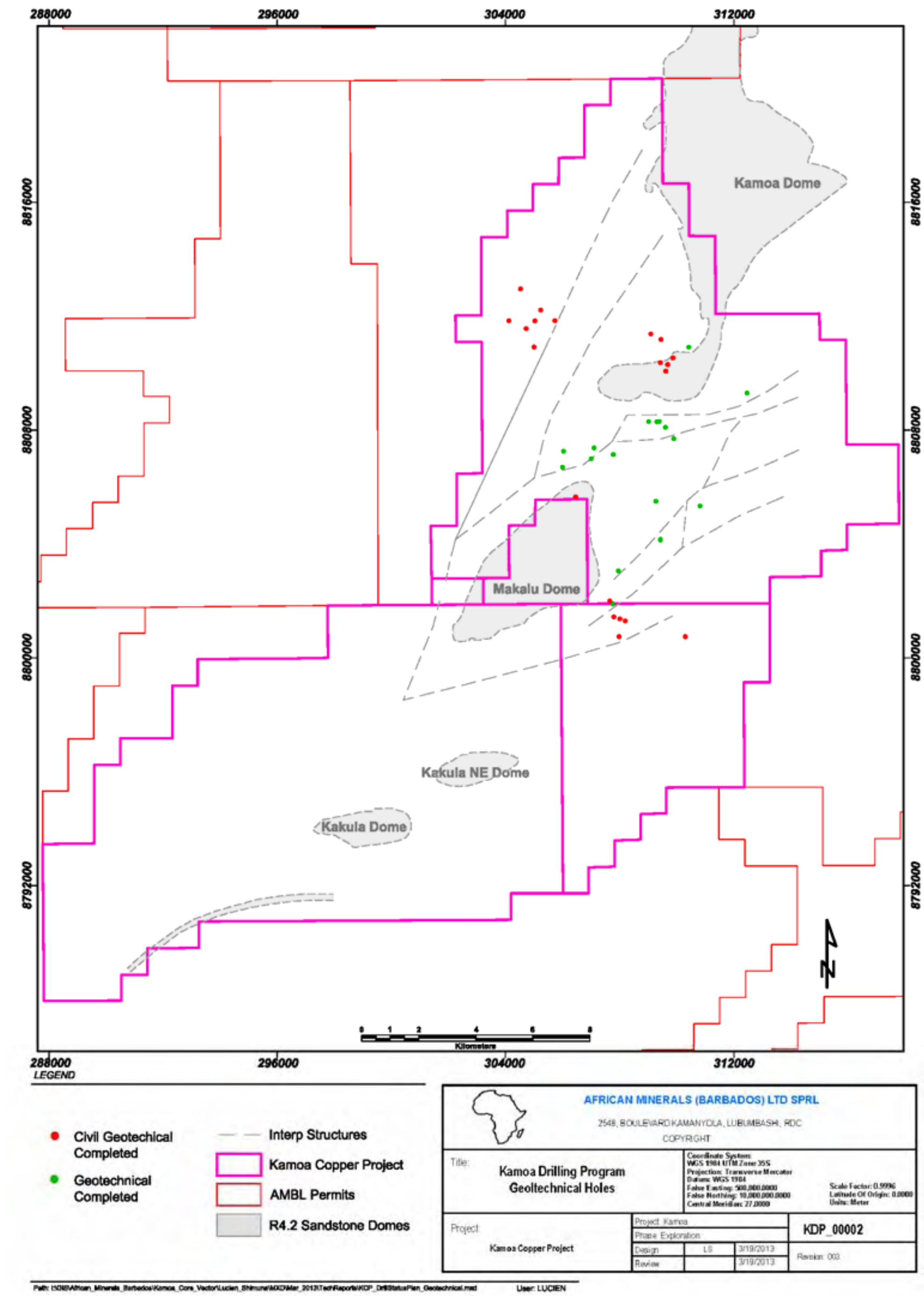
Where core is sufficiently competent to allow orientation surveys to be performed, Ivanplats collects structural information for geotechnical and geological studies. Initially the spear method was utilized. Although use of the Reflex ACT tool was initiated at drill hole DKMK_DD130, reliable readings were not obtained until drill hole DKMC_DD197, as the operators needed to gain familiarity with the instrument in order to obtain accurate measurements. The results of the program confirmed shallow to sub-horizontal dips.

10.7 Geotechnical Drilling

Since the close-out date for data included in the December 2012 Mineral Resource estimate, Ivanplats has completed an additional 20 (total depth 6,178.6 m) geotechnical drill holes. There are also 20 (total depth 358.5 m) holes identified as civil geotechnical holes.

Collar locations of geotechnical drill holes completed as at 11 March 2013 are shown in Figure 10-5.

Figure 10-4: Geotechnical Drill Collar Plan



Note: Figure courtesy Ivanplats, modified by AMEC, 2013.

10.8 Hydrogeological Drilling

Golder Associates drilled 16 hydrogeological drill holes (total depth 2,300 m) in order to provide an overview of the groundwater conditions across the concession area and to obtain baseline hydrogeological data. The location of these drill holes is shown in Figure 10-6.

Ivanplats has also completed two hydrogeology core holes (total depth 208 m) and 47 rotary air blast (RAB) hydrogeology holes (total depth 8,933.7 m).

10.9 Metallurgical Drilling

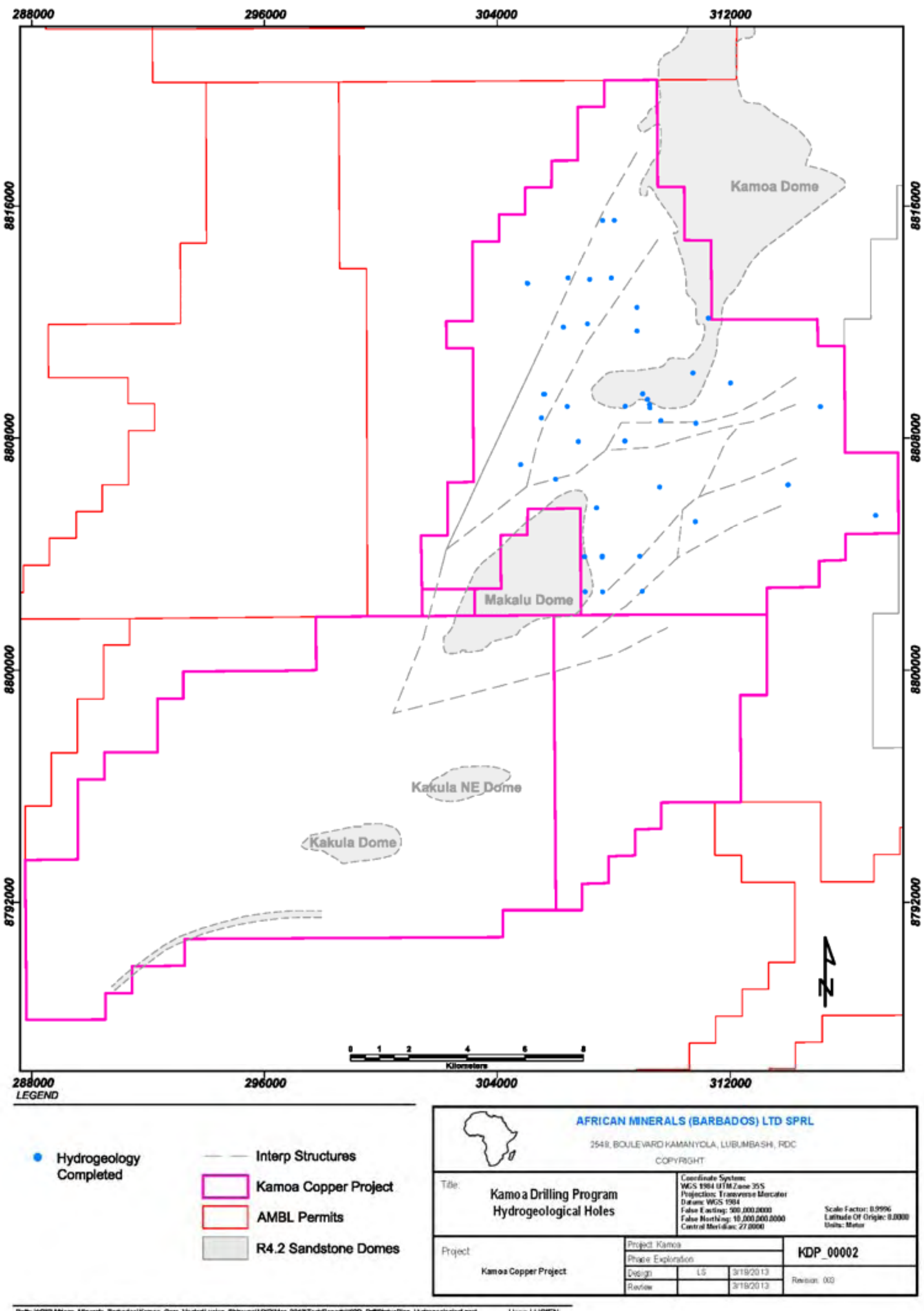
The Project database contains 23 metallurgical holes (total depth 5,282.9 m). As of 11 March, 2013, there were two metallurgical holes being drilled.

A preliminary program of metallurgical testwork was carried out by Mintek (Johannesburg, South Africa) between May and June 2010 using 12 composite samples (total of 119.8 m) from 12 core holes from the Kamoa Project. The samples were selected from drill core material mainly derived from the relatively high-grade and shallow Kamoa Sud area. In this area, supergene, mixed and hypogene mineralization occurs.

Some samples from the deeper hypogene mineralization in the eastern part of the deposit were also tested.

These samples were obtained for the purposes of gaining a general understanding of the metallurgical response of the deposit and were not intended to be used to quantify metallurgical parameters for the various geometallurgical units. The Mintek 2010 metallurgical samples were selected from available cores representing known areas within the deposit for the purpose of conducting scouting tests. These were the first metallurgical samples collected from the deposit, and the limitations of the samples were well understood at the time they were collected.

Figure 10-5: Hydrogeological Drill Hole Location Map



Note: Figure courtesy Ivanplats, modified by AMEC, 2013.

Later in 2010, samples were collected from an additional six drill holes away from the previous Kamoa Sud area. Two of the drill hole samples were taken from the Kamoa Centrale region, north of the Kamoa Dome, and four drill hole samples were taken from mineralization along the Kansoko Trend, south-east of Kamoa Sud.

The sampling performed for the Xstrata Process Support (XPS) testwork was carried out in greater detail, and with reference to more extensive drilling and a more advanced mining concept for the deposit. XPS collected samples from 34 holes totalling 258 m, which included mineralization from the Kansoko Centrale (hypogene) and Kamoa Ouest (mixed mineralization) for further metallurgical testing.

During early 2013, Ivanplats collected a suite of samples from the area of the proposed open pits to support flotation and comminution variability testwork.

The locations of drill holes completed for (or were partially used in) metallurgical testing prior to 2013 are shown in Figure 10-6. None of the samples collected in 2013 had undergone metallurgical testing at the time of issue of this Report.

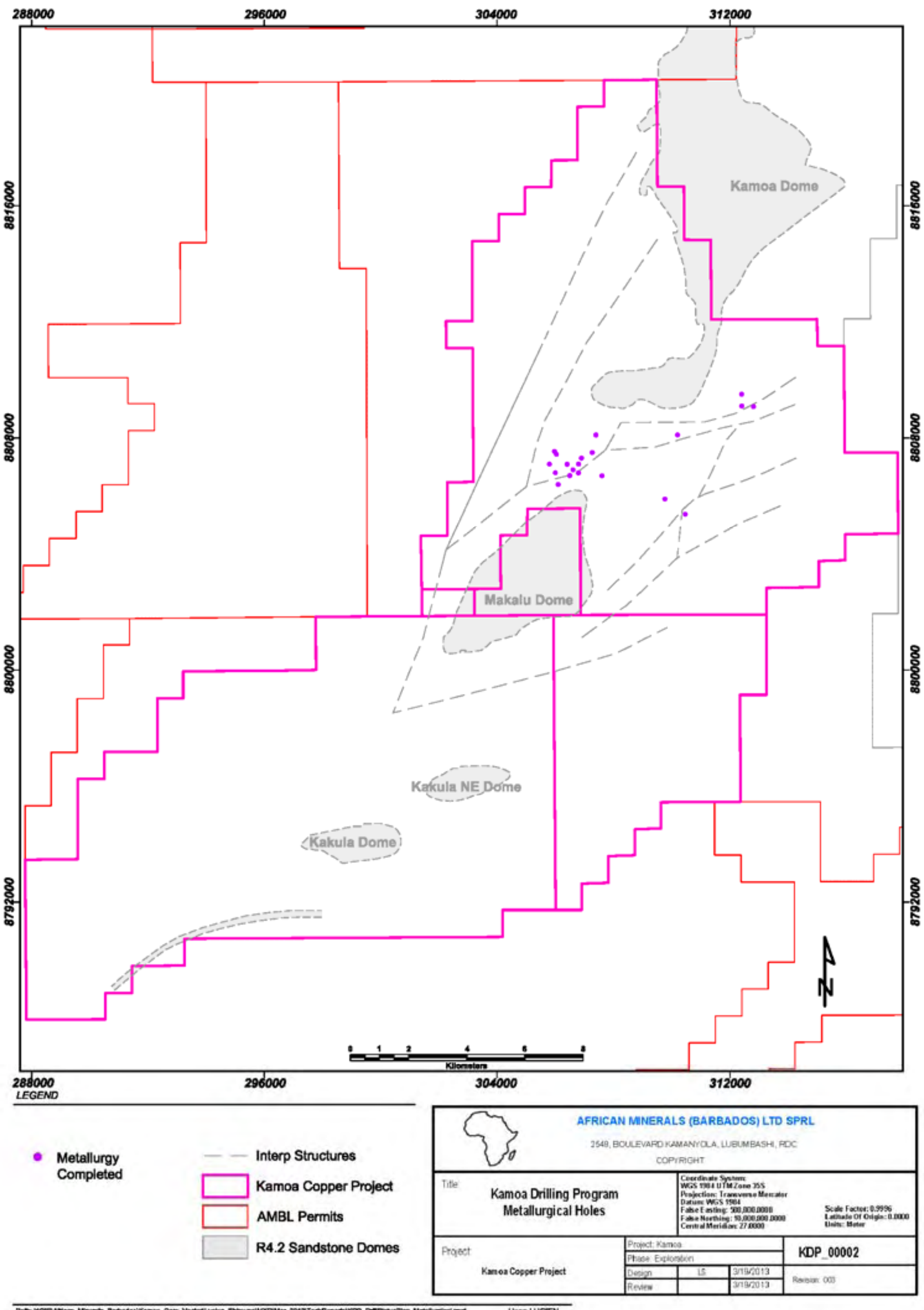
10.10 Sample Length/True Thickness

Drill intersections, due to the orientation of the drill holes, are typically greater than the true thickness of the mineralization; however, for the majority of mineralized intercepts a reduction of less than 10% in the intercept length is required to correct to true thickness (Figure 10-7). Intercepts in vertical drill holes, due to the shallow to flat dip of the mineralization, will approximate true thickness; the 151 holes that were inclined at 25° to 80° will have drilled intersections that are greater than the true thickness of mineralization. In some cases the “angle” holes have been purposely drilled sub-perpendicular to the mineralization, and for these holes the intercepted and true thickness will be similar.

10.11 Drilling Since the Mineral Resource Database Close-Off Date

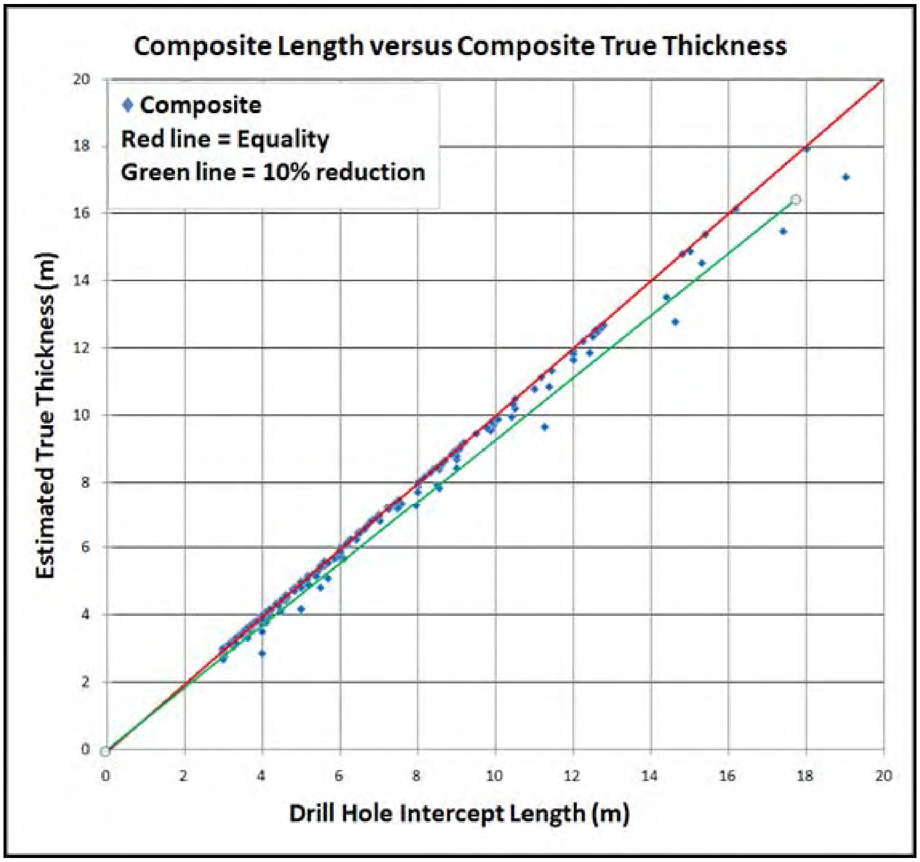
The database was closed for estimation of the updated Mineral Resources on December 10, 2012. As of March 11, 2013, Ivanplats had completed and received assays for 10 additional core holes (2,722 m drilled, 311.6 m assayed) within the resource model area. In addition, assay results are pending for 74 holes (5,850 m drilled, 2,630.5 m sampled) and two holes (417.5 m) had been abandoned. One of the abandoned holes had been partially sampled.

Figure 10-6: Metallurgical Drill Hole Location Map



Note: Figure courtesy Ivanplats, modified by AMEC, 2013.

Figure 10-7: Scatterplot of Drill Hole Intercept Length versus Estimated True Thickness (based on mineralized intercepts used for the September 2011 Mineral Resource estimate)



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, and may support potential upgrades in the resource confidence classification for some blocks from Inferred to Indicated.

Table 10-4 shows assay results and thicknesses for 12 selected selective mineralized zone (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-mineralized to >1% Cu is usually distinct, and within the mineralized 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 was used to form the SMZ.

Table 10-2: Drill Intercept Summary Table

Drill Hole ID	Easting (X)	Northing (Y)	Elevation (Z)	Depth from (m)	Depth to (m)	Intersection length (m)	Azimuth (º)	Dip (º)	Total Depth (m)	TCu (%)	TRUETHK (m)
DKMC_DD542	308799.9	8813100	1337.105	117.3	120.5	3.2	270.0	90.0	126.1	0.09	3.16
DKMC_DD026	307903.9	8812701	1193.076	239.0	242.0	3.0	56.3	89.1	284.5	0.30	2.99
DKMC_DD101	306799.8	8809500	1407.48	26.0	29.0	3.0	276.0	89.9	51.0	0.87	2.98
DKMC_DD389	309566.1	8803177	796.2034	728.0	731.0	3.0	304.2	62.0	749.0	1.26	2.87
DKMC_DD622	312320.6	8805947	302.0027	1241.7	1245.0	3.3	325.7	67.6	1252.5	2.42	3.14
DKMC_DD609	312336.2	8805434	240.1084	1288.2	1292.1	3.9	303.8	76.5	1298.0	3.02	3.76
DKMC_DD163	308600.7	8811101	1370.783	81.0	89.0	8.0	120.0	88.3	101.0	3.63	7.93
DKMC_DD606	308006.3	8807108	1162.878	285.0	295.5	10.5	57.9	87.1	311.0	4.25	10.13
DKMC_DD056	309600.6	8808404	1314.904	144.0	147.8	3.8	83.7	87.2	158.5	8.76	3.70

10.12 Comments on Section 10

The quantity and quality of the lithological, geotechnical, collar, and down hole survey data collected in the core drill programs is sufficient, in the opinion of the AMEC QPs, to support Mineral Resource estimation as follows:

- Drill intersections, due to the orientation of the drill holes, are typically slightly greater than the true thickness of the mineralization
- Drill hole orientations are generally appropriate for the mineralization style
- Core logging meets industry standards for sediment-hosted copper exploration
- Collar surveys were performed using industry-standard instrumentation
- Down-hole surveys provide appropriate representation of the trajectories of the core holes
- Core recoveries are typically excellent
- The SMZ can include both lower- and higher-grade mineralization; however, the change in grade from non-mineralized to >1% Cu is very distinct, and within the mineralized zone, grades typically remain above the 1% Cu over the entire intercept.
- No material factors were identified with the data collection from the drill programs that could affect Mineral Resource estimation.

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Witness Sampling

Ivanplats 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.

An additional government requirement is that quarter-core is also retained for witness samples. The first three core holes at Kamoa were quarter-core sampled to adhere to governmental requirements, and these witness samples have been stored in the Kamoa sample storage facility. Quarter-core sampling ceased in January 2008, and all samples subsequent to that date have been half-core samples. Should such a request be made, Ivanplats will provide either quarter core samples to the DRC Government from the archived half core or sample pulps as appropriate.

11.2 Sampling Methods

11.2.1 Geochemical Sampling

Stream-sediment samples were collected from suitable drainages, as 2 kg to 5 kg samples, and the sample location points were recorded using a GPS. Samples were subsequently dried and sieved to -150 µm at the mobile exploration base camp. An appropriate subsample was selected from the sieved sample, and the subsamples were placed into paper bags for laboratory despatch.

Soil samples were collected from the B horizon depth (30 cm to 40 cm) as 4–5 kg samples, and the sample location points were recorded using a GPS. Soil samples were transported to the appropriate mobile base camp for drying and sieving to minus 80 mesh. An appropriate subsample was selected from the sieved sample, and the subsamples were placed into paper bags for laboratory despatch.

Aircore drill samples were typically taken at the point of blade refusal, which corresponds to the base of the drill hole. One sample was taken per drill hole, and each sample typically weighed about 2 kg to 5 kg. A representative sample is retained for each hole and is placed in chip trays that are stored in the Kolwezi storage facility.

11.2.2 RC Sampling

Reverse circulation (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. All reject material was disposed of in the field. During sampling, duplicate samples were typically taken at a frequency of one duplicate for each group of 20 samples.

Homogenization was achieved by mixing the 1 kg samples in clean plastic bowls, and then splitting the homogenized sample using a single-stage Jones riffle-splitter to produce one sample for assay and one composite sample for reference. The 1 kg sample for analysis was submitted to the Kolwezi sample preparation facility. Prepared and reference samples were placed in courier boxes for submission to the analytical laboratory, and for submission to the DRC government geological department as part of the obligation to supply witness samples.

11.2.3 Core Sampling

Drill holes DMAK_DD001 to DMAK_DD003 were subject to quarter-core sampling in 2006; quarter-core sampling was discontinued in January 2008.

The current core sampling procedure from January 2008 to date is as follows.

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

Pre-February 2010, determination of the sample intervals took into account lithological and alteration boundaries. The entire length of core from 4 m (or one core-tray length whichever is convenient) above the first presence of mineralization and/or the mineralized zone 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 mineralized basal diamictite was conducted as follows:

- The mineralized 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. (Note that there is a 3 m shoulder left above the first visible sign of copper mineralization in each drill hole)
- After March 2011, 9 m composite samples were collected in the hanging wall and analysed by Niton. The results are used to characterize the geochemisty of the hanging wall material.

- 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 down hole, with the half section that contains the core orientation line retained in the core trays for geological logging and record purposes. The half-core along the right hand side of the projected orientation lines is sampled and sent to the preparation laboratory.
- Oxide-zone samples are split using a palette knife, and the same sample protocol that is used for un-oxidized 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 nominal 2 mm using jaw crushers. A quarter split (500 g to 1,000 g) is pulverized 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 prevent oxidation.

Samples collected in 2013 consist of a mixture of whole PQ and half PQ core.

11.4 Density Determinations

Specific gravity measurements were performed using a water immersion method by Ivanplats personnel supervised by either Ivanplats' personnel or African Mining Consultants' geologists. Samples were conventionally weighed in air and then in water. As a check, African Mining Consultants took carefully-measured cylinders of dry core, calculated a volume and weighed them dry to arrive at a specific gravity. The cylinders were then weighed in water and the density calculated by the same method as for all other samples. The relative difference was <2%. The scales were regularly calibrated using set weights, and also using clean quartz crystals, to give a constant specific gravity of 2.64 for the quartz crystals.

A total of 10,111 specific gravity measurements were performed on samples taken from drill core.

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 Vancouver, British Columbia, acted as the check laboratory for drill core samples from part of the 2009 program and for 2010 through 2012 drilling. ALS Chemex is ISO:9000:2008 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 pulverizers. The laboratory is managed by Ivanplats personnel and was monitored by Richard Carver of GCXplore Ltd. between 2006 and 2009.

A 500 g to 1,000 g RC sample is reduced to >90% -75 µm, using the LM2© puck and bowl pulverizers. Two samples are subsequently split from the pulp; one 30 g sample is sent for analysis, and the second 100 g split is retained as a reference sample, and is stored in Kolwezi. Aircore samples from the 2008 drilling were processed in a similar manner to the RC samples.

All drill-core samples collected prior to November 2010 were processed by the Kolwezi facility; subsequently they have been processed at the new Kamoa site facility. Core samples are delivered from the core logging facility to the sample preparation facility by truck. 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 the sample forms are signed. Sacks are not opened until sample preparation commences.

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 (500g to 1,000 g) is pulverized in the same manner as the RC samples. 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, 30 g for Niton analysis, and approximately 80 g of pulp is retained as a reference sample.

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 pulverized material are then labelled and bagged for shipment by air to Western Australia. From 2010, Ivanplats has been weighing the pulp samples and records the weight prior to shipping.

Certified reference materials and blanks are included with the sample submissions. Ivanplats commissioned a second, dedicated sample preparation facility on-site at Kamoa in November 2010. The equipment at the facility includes a TM Terminator Jaw crusher, Labtech Essa LM-2 pulverizer and a riffle splitter. This facility has been in use since DKMC_DD209.

11.7 Sample Analysis

11.7.1 Genalysis

In 2004, all soil and stream-sediment samples were analyzed by Genalysis. The -50 µm sieved sample, as received by the laboratory, was pulped to 90% passing -75 µm. The analytical method used for Au, Pt, and Pd involved reading by inductively coupled plasma-mass spectrometer (ICPMS) following a 25 g lead collector fire assay

(Genalysis method FA25/MS). The elements Cu, Co, Ni, Fe, Mn, Ag, As, Zn, and Bi were determined using inductively coupled plasma (ICP) optical emission spectroscopy (OES) following a hydrofluoric–nitric–perchloric–hydrochloric acid digestion (Genalysis AT/OES method).

From January to June 2005, soil and stream-sediment samples were analyzed for a suite that ranged from five to 16 elements, depending on the analytical request from the geologist. Genalysis performed a 10 g aqua-regia digest with inductively coupled plasma optical emission spectroscopy/mass spectrometer (ICP-OES/MS) finish, analyzing for elements that could include Cu, Co, Ni, Fe, Mn, Ag, As, Zn, Bi, Mo, Ba, Ti, Cr, Pb, and Au.

11.7.2 Ultra Trace

For all soil and stream-sediment samples, a 4 g sample was taken from the pulps, and digested using an aqua-regia digest. Samples were analyzed for Cu, Co, Ni, Zn, As, Mo, Au, Ag, Ba, Bi, Cr, Fe, Mn, Pb, and U via ICP-OES/MS.

Aircore, RC and diamond drill hole samples for all programs from 2008 to February 2009 were analyzed 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 analyzed 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 OES) or ICP302/AR001 (ICP-MS).

Diamond drill samples from January 2010 onward were also analyzed for acid-soluble copper (ASCu) using a 5% sulphuric acid leach method at room temperature for 60 minutes. AMEC recommends that samples obtained in 2008 and 2009 be submitted for ASCu analysis. As of 11 March 2013 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 analyzed for Cu, S (Ultra Trace method MA201, ICP-OES), and As (Ultra Trace method MA202, ICP-MS). Samples within the mineralized basal diamictite were analysed for Cu, Fe, S (Ultra Trace method MA201), Ag, and As (Ultra Trace method MA202).

Few samples were originally submitted for ASCu; however, recently these samples have been submitted to obtain ASCu results (Ultra Trace method SA201–SA100, 5%

sulphuric acid leach). Samples obtained from the due diligence drill holes were submitted for ASCu analysis.

Detection limits for elements published by Ultra Trace for the ICP-MS method are summarized in Table 11-1 and in Table 11-2 for the ICP-OES method.

Table 11-1: Analytical Detection Limits for Ultra Trace ICPMS Analytical Method

Element (Detection Limit)						
Ag (0.05 ppm)	As (0.2 ppm)	Ba (0.05 ppm)	Bi (0.002 ppm)	Cd (0.1 ppm)	Ce (10 ppb)*	Co (0.2 ppm)
Cs (20 ppb)	Cu (0.5 ppm)	Dy (10 ppb)*	Er (10 ppb)*	Eu (2 ppb)*	Ga (0.2 ppm)	Gd (0.05 ppb)*
Hg (0.01 ppm)	Ho (5 ppb)*	In (5 ppm)	Ir (5 ppb)*	La (10 ppb)	Li (0.1 ppm)	Lu (5 ppb)*
Mo (0.1 ppm)	Ni (1 ppm)	Nd (10 ppb)*	Pd (10 ppb)*	Pb (1 ppm)	Pr (5)	Pt (5 ppb)*
Sn (0.2 ppm)	Sr (0.1 ppm)	Tb (5 ppb)*	Te (0.1 ppm)	Th (10 ppb)	Tl (10)*	Tm (5 ppb)*
U (10 ppb)*	W (0.1 ppm)	Y (10 ppb)*	Yb (10 ppb)*	Zn (1 ppm)	--	--

Table 11-2: Analytical Detection Limits for Ultra Trace ICPOES Analytical Method

Element (Detection Limit)				
Ag (1 ppm)	Al (10 ppm)	As (5 ppm)	Ba (1 ppm)	Be (1 ppm)
Ca (10 ppm)	Cd (1 ppm)	Co (2 ppm)	Cr (5 ppm)	Cu (1 ppm)
Fe (10 ppm)	K (20 ppm)	Li (10 ppm)	Mg 910 ppm)	Mn (1 ppm)
Mo (2 ppm)	Na (10 ppm)	Ni (1 ppm)	P (10 ppm)	Pb (5 ppm)
S (10 ppm)	Sc (1 ppm)	Sr (2 ppm)	Sn (5 ppm)	Ti (1 ppm)
V (2 ppm)	Y (5 ppm)	Zn (1 ppm)	Zr (1 ppm)	--

11.8 Quality Assurance and Quality Control

11.8.1 Blanks

Five materials, BLANK2005, BLANK2007, BLANK2008, BLANK2009, and BLANK2010 have been used for the Kamoa drilling. 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-mineralized. In the opinion of Ivanplats' 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 programs. BLANK2010 has been used in the 2010 through 2013 drill programs. One blank per 20 samples was inserted prior to the samples being pulverized.

11.8.2 Duplicates

One duplicate was taken every 20 samples. This was achieved by taking a second 1 kg split of the drill interval being sampled for the reverse circulation drilling. For the diamond drilling, a preparation duplicate was created for every 20th sample by taking a second split following the crushing stage of the sample preparation. AMEC also compiled Ultra Trace's QC control duplicates (same-pulp same-batch laboratory duplicates) from the laboratory reports.

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. For the 2005 RC program, these CRMs were inserted in Zambia by an African Mining Consultants representative, who placed the CRMs in sample number positions as indicated by the field geologist. Since 2006, CRMs have been inserted by Ivanplats personnel in Kolwezi, and since November 2010 have been inserted by Ivanplats personnel at the Kamoa Project site.

11.9 Databases

Project data are stored in various digital files. Geological logs, collar, and down hole survey data were hand-entered at the Kolwezi office into Word and Excel files. Data are currently entered at the Kamoa (site) office, and no longer in Kolwezi. Assay data are imported directly from electronic files provided by the assay laboratory.

Original hard-copy data are stored in the Ivanplats' Kamoa site office. 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

Transport and security procedures from the sample site to the sample preparation facilities and thence to the laboratory are discussed in Section 11-5.

Paper records are kept for all assay and QA/QC data, geological logging and density information, and down hole and collar coordinate surveys. All paper records are filed by drill hole for quick location and retrieval of any information desired. Assays, down hole surveys, and collar surveys are stored in the same file as the geological logging information. In addition, sample preparation and laboratory assay protocols from the laboratories are monitored and kept on file.

RC and aircore chip trays are stored in the Kolwezi compound. 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. The RC and aircore samples remain at the Kolwezi storage facility.

11.11 Comments on Section 11

The AMEC QPs note:

- A description of the geology and mineralization of the deposits, which includes rock types, geological controls and widths of mineralized zones, is given in Section 7.
- A description of the sampling methods, location, type, nature, and spacing of samples is included in Section 9 and Section 10, with appropriate location plans to show the density of sampling and areas sampled.
- A description of the drilling programs, including sampling and recovery factors, are included in Section 10. Review of these programs indicates that there are no issues that could affect Mineral Resource estimation.

- In the absence of detailed comparisons between RC and core samples, in the AMEC QPs' opinion, Ivanplats made the correct decision to exclude RC samples from the dataset used for Mineral Resource estimation for both the estimate supporting the 2012 PEA and for this current update.
- The core drill sample results have outlined a large area of continuous mineralization on the flanks of the Kamoa and Makalu domes. The exploration is as yet incomplete, with only part of the area having been drilled at the 800 m spacing that the AMEC QPs consider to be the minimum spacing required at present to estimate Inferred Mineral Resources at Kamoa.
- Drilling to date indicates that the size of the sampled area is representative of the distribution and orientation of the mineralization. Holes have been drilled sub-perpendicular to the mineralization, and local in-fill drilling has been completed to 400 m, 200 m, and 100 m spacings.
- A summary of typical drill hole intercepts is shown in Table 10-4 and Figure 7-5 to Figure 7-7 in Section 7 using a 1.0% Cu cut-off grade. The intercept table confirms that sampling was representative of the copper grades in the deposits, reflecting areas of higher and lower grades.
- Data validation of the drilling and sampling program is discussed in Section 12, and the discussion includes review of database audit results, independent sampling, and QA/QC.

In the opinion of the AMEC QPs, the sampling methods acceptable, consistent with industry-standard practice, and adequate for Mineral Resource 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 mineralization changes in the core, are typical of sample intervals used for Copperbelt-style mineralization in the industry
- Samples are taken for assay depending on location, stratigraphic position, and observation of copper mineralization
- Sampling is considered to be representative of the true thicknesses of mineralization. Not all drill core is sampled; sampling depends on location in the stratigraphic sequence and logging of visible copper-bearing minerals

- The density determination procedure is consistent with industry-standard procedures. There are sufficient density determinations to support the density values utilized to estimate the resource tonnage
- Preparation and analytical procedures are in line with industry-standard methods for Copperbelt-style copper mineralization, and suitable for the deposit type
- A QA/QC program comprising blank, CRM, and duplicate samples is used on the Project. QA/QC submission rates meet industry-accepted standards
- Sample security has relied upon the fact that the samples were always attended or locked in the on-site sample preparation facility. The chain-of-custody procedure consists of filling out sample submittal forms that are sent to the laboratory with sample shipments to make certain that all samples are received by the laboratory
- Current sample-storage procedures and storage areas are consistent with industry standards.

12.0 DATA VERIFICATION

12.1 AMEC Verification

12.1.1 2009 Verification

AMEC reviewed the data available to support Mineral Resource estimation as of end June, 2009. Reviews included checking of collar co-ordinates, drill collar elevations and orientations, down-hole and collar survey data, geological and mineralization logging, and assay and density data. No significant errors were noted that could affect Mineral Resource estimation.

Reviews were also performed on CRM, blank and duplicate analytical data and screen tests; no biases that could affect Mineral Resource estimation were observed.

12.1.2 2010 Verification

AMEC reviewed the data available to support Mineral Resource estimation as of end July, 2010 (Long, 2010, Reid 2010b). Reviews included checking of collar co-ordinates, drill collar elevations and orientations, down-hole and collar survey data, geological and mineralization logging, and assay and density data. No significant errors were noted that could affect Mineral Resource estimation.

Reviews were also performed on CRM, blank and duplicate analytical data and screen tests, no biases that could affect Mineral Resource estimation were observed.

AMEC suggested changes to improve the mineral abundance logging in order to reduce overestimation of sulphide mineral abundances.

12.1.3 2011–2012 Verification

Since September 2011, AMEC has conducted monthly audits of the additional collar, down-hole survey, geology and assay data collected to ensure the data quality has been maintained. Any discrepancies noted were supplied to site, and corrections to the database were implemented. In December 2012, it was decided to perform quarterly audits from that point onwards.

AMEC reviewed the data available to support Mineral Resource estimation as of end December, 2012. No significant errors were noted that could affect Mineral Resource estimation.

Reviews were also performed on CRM, blank and duplicate analytical data and screen tests, no biases that could affect Mineral Resource estimation were observed.

AMEC checked the drill data collected subsequent to August 2012 used to support the updated Mineral Resource estimate for data integrity. Following the checks, AMEC concluded that the error rates for survey, assay and critical geological databases were within acceptable bounds (1%) for the 2012 updated database.

12.2 QA/QC Review

AMEC reviewed the QA/QC data collected from 2008 to August 2012 (Reid, 2012). An updated memorandum by AMEC (Yennamani, 2012) discussed the QA/QC data reviewed from August 2012 to December 2012.

12.2.1 Screen Tests

Screen tests to monitor crusher output before splitting and pulverizer 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

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. African Mineral 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'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. AMEC identified results that differed from the certified value by more than 25% of the certified value. There are 34 CRM samples out of 3,251 samples that show such levels of disagreement.

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

AMEC excluded these outliers, and 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 reviewed 109 CRM results obtained from August to December 2012 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 QPs, with the exception of 45P, 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

Previous Check Assays

Check assays were performed prior to 2010 using a different, multi-acid digestion method; check assays were compared with the original values, and found to be acceptable at grades above 0.1% Cu. Results confirm that significant copper mineralization is hosted in sulphides, but copper is also occurring in other minerals (less soluble in aqua-regia) in the more weakly mineralized samples. Comparison of Genalysis to Ultra Trace results 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.

2010 ALS Chemex Check Assays

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 were assayed for total and acid-soluble Cu, Co, Fe, Mn, As, Pb, Zn and S (two methods). The 767 sample pulps submitted included 22 blanks and 39 CRMs.

The blanks show sufficiently low values, reflective of the detection limits. The CRMs indicate ALS Chemex results have good accuracy for copper, a low bias for Pb and Zn, and somewhat erratic results for sulphur. There are some significant differences at near-detection grades, caused by higher detection limits of the ALS Chemex results; below-detection results were set equal to half the detection limit. The lower detection limit of the Ultra Trace results produces a negative intercept artefact, in cases where a significant number of results are below the Chemex detection limit.

Results for elements other than Pb had no outliers, or outliers that had no significant impact on the linear fit calculation. Linear regression parameters were calculated by the Reduction to Major Axis method. These all showed agreement between Ultra Trace and ALS Chemex within 10% (slope 1 ± 0.1), and all except arsenic agree within 5%.

2011 ALS Chemex Check Assays

No samples had been submitted to a check assay laboratory since 2010. In late 2011 AMEC provided Ivanplats with a list representing a random 5% selection of samples within and adjacent to the SMZ. These samples were submitted to ALS Chemex (Vancouver, Canada) and were assayed for total and acid-soluble Cu, Co, Fe, Mn, As, Pb, Zn and S (two methods). The 1,053 sample pulps submitted included 102 blanks and 130 CRMs.

The results for the blank samples do not show signs of sample contamination and are typically below or near detection limits with the exception of Fe. The results for Fe range from around 0.5% to just over 1.0 % Fe; it is likely the blank sample contains low levels of Fe.

The CRMs indicate ALS Chemex results have acceptable accuracy for Cu and Co. The results for Fe tend to be biased low for CRMs below 6% Fe and accurate for higher grades. Lead is typically biased low, Zn is acceptable, and S is biased low for CRMs over 10%. This is apparent in the scatterplots for S assays.

Comparisons did not have outliers that could significantly impact on the linear fit calculation. Linear regression parameters were calculated by the RMA method. These all showed agreement between Ultra Trace and ALS Chemex within 10% (slope 1 ± 0.1), and all except S agree within 5%. This is very good agreement.

2012 ALS Chemex Check Assays

In late 2012 AMEC provided Ivanplats with a list representing a random 5% selection of samples within and adjacent to the SMZ. These samples were submitted to ALS Chemex (Vancouver, Canada) and were assayed for total and acid-soluble Cu, Co, Fe, Mn, As, Pb, Zn and S (two methods). The 1,053 sample pulps submitted included 47 blanks and 114 CRMs. The samples had not been analyzed for Mn, Pb and Zn by Ultra Trace.

The results for the blank samples do not show signs of sample contamination and are typically below or near detection limits with the exception of Fe. The results for Fe range from around 0.5% to just over 1.0 % Fe, it is likely the blank sample contains low levels of Fe.

The CRMs indicate ALS Chemex results have acceptable accuracy for Cu with the exception of AMIS0120 where ALS Chemex is biased 9.9% low. This is noted in the Cu scatterplots. ASCu results for AMIS050 indicate acceptable accuracy. The results for Fe are biased low for all CRMs, especially for grades below 6%. Sulphur is biased low for CRMs over 10%. This is apparent in the scatterplot for S assays.

A few sample pairs considered as outliers were removed from the analysis. Linear regression parameters were calculated by the RMA method. These all showed agreement between Ultra Trace and ALS Chemex within 10% (slope 1 ± 0.1), and all except As agree within 5%. This is very good agreement.

12.2.4 Duplicate Assays

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 evaluated the duplicate samples by calculating the Absolute Value of the

Relative Difference (AVRD), equal to two times the absolute value of the pair difference divided by the pair mean:

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

Duplicate pairs of this type have good precision if 90% of mineralized pairs (i.e. samples with grades well above the analytical detection limit and at or above the lowest probable mineralized 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 finds the assay precision is acceptable for Mineral Resource estimation.

12.2.5 Blanks

AMEC reviewed the results for 1,882 blank samples submitted for analysis. In the opinion of Ivanplats' 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 considers that there is no significant risk to the Mineral Resource estimate.

12.2.6 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-mineralized 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.

AMEC notes that there is a need to correlate ASCu with metallurgical recovery, particularly in partially leached zones.

12.3 Site Visits

Site visit dates and the scopes of personal inspection by the QPs are discussed in Section 2.

12.3.1 Field Drill Collar Check

Field drill collar checks were completed as follows:

- In 2009 AMEC used a hand-held GPS unit to check the coordinates of five drill holes in the field.
- During the 2010 visit AMEC used a hand-held GPS and Brunton Compass to check the coordinates and orientation of six drill holes in the field.
- AMEC used a hand-held GPS unit to check the location of 10 drill holes during the February 2011 site visit, 15 holes during the November 2011 site visit, and 11 holes during the November 2012 site visit.

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

12.3.2 Core Storage

Drilling was being conducted during the 2010 AMEC 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.

In 2010, a new core-logging facility and new secure core-storage facility were constructed at the Kamoa camp. As of July 2010, all new core samples are stored at the new facility. Mineralized core prior to 2010 has now been moved to this new facility; the historical RC and aircore samples remain at the Kolwezi storage facility. Figure 12-1 and Figure 12-2 show the logging facility and core storage respectively.

The AMEC QPs are of the opinion that the storage and logging facilities are acceptable and meet industry standards.

12.3.3 Inspection of Drill Core

The following core holes were examined during the 2009 visit:

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

The following core holes were examined during the 2010 visit:

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

Figure 12-1 On-Site Core-Logging Facility

Note: Photograph courtesy AMEC, 2011

Figure 12-2 On-Site Core-Storage Facility

Note: Photograph courtesy AMEC, 2011

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

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

Logging details were noted, in general, to match the features that AMEC 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 site visit, AMEC 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's Confirmation of Copper Grades

In 2009, AMEC 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'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 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 selected 22 sample intervals from eight drill core boxes. The half core in the boxes was re-sawn, and quarter-core samples were taken by African Mining Consultants under AMEC'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 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 selected 11 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'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-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 selected eight 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's direction, and submitted, along with CRMs and blanks, to Ultra Trace in Australia. AMEC's samples were found to be comparable to the original Ultra Trace results; AMEC's Cu results were 4% lower than the original assays while the ASCu results were 2% higher.

In November 2012, AMEC selected eleven 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's direction, and submitted, along with CRMs and blanks, to Ultra Trace in Australia. Ultra Trace analysed the samples and then shipped the samples to ALS Chemex in Vancouver, Canada. Results from ALS Chemex have not been received. Ultra Trace's witness sample results averaged 10% lower than the original Ultra Trace 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, and during 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 Ivanplats 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, down hole direction, and actual measurement.
- Geotechnical database: The Kamoa geotechnical database is 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 QPs, the data verification programs undertaken on the data collected from the Project support the geological interpretations, and the analytical and database quality. Therefore, the collected data can support Mineral Resource estimation. Principal findings from the data verification are as follows:

- The accuracy of the current surface topography is estimated to be ±12 m. Ivanplats has obtained high resolution topgraphic data based on a LiDAR survey conducted in 2012. AMEC compared the collar elevation data to this data and found 62 drill holes with collar elevation discrepancies of over three metres. A list of these drill holes was sent to Ivanplats for investigation
- Sample data collected adequately reflect deposit dimensions, true widths of mineralization, and the style of deposit
- Drill collar data were verified by comparing database values to records obtained from the survey contractor
- Verification of down-hole survey data was completed
- Excessively high error rates were noted in some of the non-crucial geology fields and density data in 2010. This was not noted in the 2011 or 2012 audits
- It is apparent Ivanplats is not reviewing the CRM results in “real time” for mislabelling. For a QA/QC program to be effective it is necessary to review the results frequently to determine if corrective action must be taken. Ivanplats is using its independent geochemical consultant Richard Carver to monitor CRM results versus their certified values
- The quality assurance program for the core drilling on the Project demonstrates sufficient accuracy and precision of the copper assays for use in estimating Mineral Resources for copper
- 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

- AMEC's independent sampling of 73 drill core intervals, with assaying by independent laboratories that were selected by AMEC confirms the Cu and Co grades reported by Ivanplats
- AMEC provided Ivanplats with a list representing a random 5% selection of samples within and adjacent to the SMZ. These samples were submitted to ALS Chemex for check analysis. Appropriate numbers of blanks and CRMs were included with this submission. Good agreement was obtained between UltraTrace, the primary laboratory, and ALS Chemex
- The geotechnical data collection is acceptable at this stage of the project, but there are a number of areas that will require additional data collection as the project continues, such as RQD measurements, core orientations, database maintenance, and improvements to detailed logging.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Metallurgical Testwork

13.1.1 Overview of Metallurgical Testwork Supporting 2012 PEA

The first metallurgical testwork on Kamoa mineralization was carried out in 2010 using drill cores from the shallow, Kamoa Sud area of the deposit. The samples were subjected to comminution and flotation tests at Mintek laboratories in Johannesburg.

Comminution tests indicated that the mineralized rock is competent with respect to SAG milling, that the Bond ball mill work index results are modest and in the range of 14 to 16 kWh/t and that the mineralization is moderately abrasive. Due to milling efficiency considerations, mineralized materials with these properties, and at treatment rates of less than about 6 Mt/a, are best processed in crush/ball mill circuits rather than SAG circuits. These comminution results show that the Kamoa mineralization is moderately harder than typical Copperbelt ores.

The Mintek flotation testwork showed that the mineralization was amenable to treatment by conventional sulphide flotation but with the proviso that a significant amount of regrinding is required. Flotation recoveries were lower than expected for Copperbelt ores due to a non-floating copper sulphide population locked in silicates at sulphide phase sizes of 10 µm or finer. The economic copper minerals include chalcopyrite, bornite and chalcocite.

Although this phase of the testwork was preliminary, it did indicate that an effective flotation flow sheet will include roughing, cleaning and multiple stages of re-cleaning after fine regrinding.

A re-cleaner concentrate containing 27% Cu, 15% Fe, 17% S and 26% SiO_2 was produced at a copper recovery of 79% on a preliminary master composite sample obtained from the mixed supergene and hypogene zones.

Subsequent testwork in 2011 at Xstrata Process Support (XPS) laboratories in Sudbury, Canada, on new and more representative samples, has shown that the ultimate Mintek flow sheet was an appropriate starting point for further optimisation work. Consequently higher concentrate grades have been achieved at improved recoveries. The XPS work forms the basis for the 2012 PEA flow sheet concept.

The best XPS results on hypogene mineralization were achieved in a circuit utilising two regrind stages, one to 15 µm and the second to 10 µm. The circuit achieved a copper recovery of 85.4% at a grade of 32.8% Cu.

13.1.2 Mintek Testwork Phase 1 (2010)

The Mintek flotation testwork was the first program conducted on mineralization from the Kamoa deposit. As such the program was intended to be a basic investigation of typical mineralized material response rather than a definitive test series. The program commenced with a focus on maximizing rougher recovery using a simple reagent scheme (collector plus frother) and by varying rougher grind conditions and flotation pH.

The program then progressed to investigating cleaning options including the use of dispersants and depressants for gangue minerals and the benefits of regrinding for improving concentrate grades. The program ceased when the concepts were proven to the point of making concentrates grading greater than 25% Cu.

Samples were selected to represent what were the three important mineralized material types at the time. These mineralized types were 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 at the time. Sample selections were made from core already drilled, logged crushed and sub-sampled for assay. Drill hole collar locations for the drilling used in metallurgical sampling were included in Figure 10-7.

The summary head analyses of the Mintek test composites are provided in Table 13-1.

Table 13-1: Mintek Phase 1 Flotation Testing Composites – Head Assays

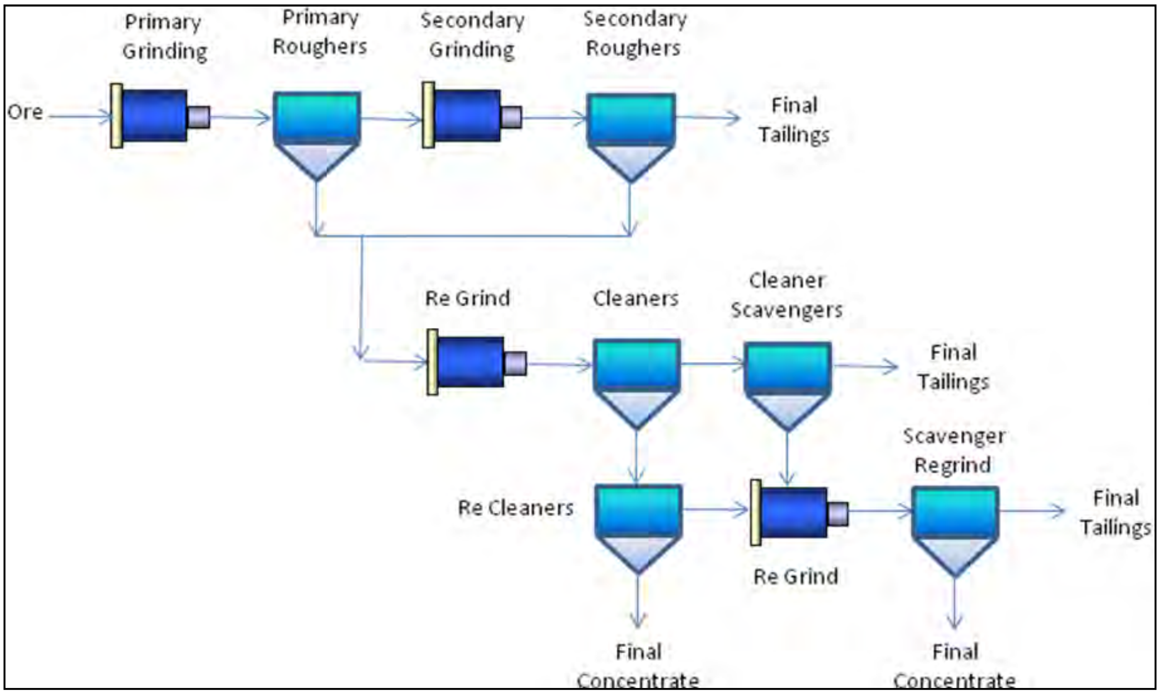
	Units	Master Composite	Supergene	Mixed	Hypogene
Total Cu	%	2.97	3.3	3.63	1.68
Total S	%	1.9	1	3.1	3.3
Al	%	4.5	3.9	3.5	4.1
Si	%	18.6	17.7	17.3	18.9

Flotation Testing and Results

Rougher plus scavenger flotation was typically able to recover between 80 and 85% of the copper when grinding (in one or two stages) was taken down to 90% -75 µm. The reagent scheme used was simple and consisted of a xanthate and a polyglycol frother, and no pH adjustment was carried out.

An MF2 rougher flotation scheme (mill-float-mill-float) achieved slightly higher recoveries than a typical MF1 arrangement and was chosen as the basis for cleaning trials. Cleaning of concentrates after regrinding to 20 to 30 µm resulted in concentrate grades in excess of 30%, but at only modest recoveries with the best overall result being 32% Cu at 73% recovery. A flowsheet (Figure 13-1) which included a second stage of regrinding on middlings streams, achieved the results shown in Table 13-2.

Figure 13-1: Dual Regrind Circuit Flow Sheet



Note: Figure courtesy Mintek 2010

Table 13-2: Dual Regrind Circuit Results

	Mass Recovered	%Cu	% Cu Recovery
Rougher Concentrate	45.9	6.3	91.4
Re-cleaner Concentrate	4.9	39.6	61.8
Scavenger Cleaner Concentrate	4.5	12.9	17.8
Combined Concentrates	9.3	27.0	79.7

The dual regrind circuit provided the highest grade and recovery combination of the Phase 1 Mintek test program.

Phase 1 continued with a set of rougher variability tests on the three individual composites, Hypogene, Mixed and Supergene. The results were similar to those achieved on the master composite.

13.1.3 Mintek Testwork Phase 2 (2010–2011)

Variability Sample Testing

New samples were sourced for this round of testwork from the 2010 drilling program. The samples were collected from a wider range of locations (refer to Figure 10-7 for the locations of the metallurgical drill holes).

These samples were tested using a simplified MF2 and cleaner flowsheet. Recoveries to re-cleaner concentrate averaged only 66% for the supergene samples and 81% for the hypogene. Concentrate grades for the supergene averaged 32% Cu, but the hypogene was poor at 17% Cu.

The tests employed a relatively simple “MF2” flow sheet which had been developed in earlier work on the master composite. It comprised milling to 80% passing 75 µm, followed by rougher flotation and two stages of concentrate cleaning. The rougher tails were then reground for 20 minutes and subjected to a scavenger flotation stage.

The tests were deliberately un-optimised to show the variability of response. The outcomes showed that mineralization from across the Kamoa deposit responds 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 recovery. The low hypogene concentrate grades confirmed that additional regrinding is necessary to achieve target.

Variability Sample Composite Testing

Two composites were prepared from the set of variability samples. The first was the Hypogene Chalcopyrite Composite and the second was the Hypogene Bornite Composite. Partially optimised MF2 flotation test outcomes are summarised in Table 13-3 for each of the composites.

Table 13-3: Mineralized Material Composite Flotation Result, MF2

Hypogene Chalcopyrite Comp				Hypogene Bornite Comp			
Rougher		Cleaner		Rougher		Cleaner	
% Cu	Rec Cu	% Cu	Rec Cu	% Cu	Rec Cu	% Cu	Rec Cu
10.2	92.6	28.4	81.6	8.4	92.2	40.8	82.7

These results are both superior to the Phase 1 result achieved with the Master Composite, confirming both the appropriateness of the flowsheet concept and the potential for further improvement with continued testing.

13.1.4 XPS Testwork to August 2012

Flotation testing was shifted to XPS laboratories in Sudbury, Ontario, Canada during 2011.

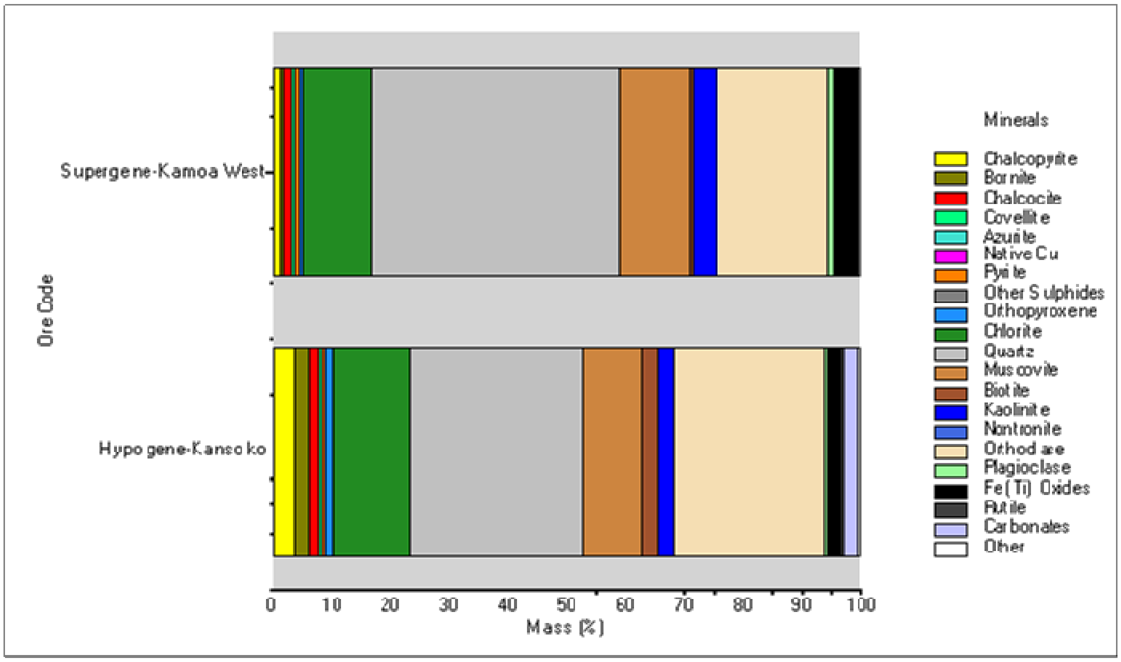
Geological progress in 2010 expanded the Kamoa copper resource with the majority of the new intersections being in hypogene mineralization. New samples of hypogene, supergene and oxide mineralization were selected for the XPS program with the primary focus being on new testwork on hypogene material.

13.1.5 Mineralogy

QEMSCAN mineralogical analysis and parallel chemical assays were performed on size fractions from representative composite feed samples after grinding to 75 µm P80. QEMSCAN was developed by the CSIRO in Australia and is an automated electron microscope based mineralogical analysis tool able to identify minerals at a resolution of 1 μm^2 or finer. The technology uses a combination of back-scattered electron (BSE) brightness, and X-ray spectrum analysis (energy dispersive spectrophotometry, EDS) to identify minerals. The system automatically analyses thousands of mineral grains per sample in detail, and provides mineralogical results of high statistical relevance.

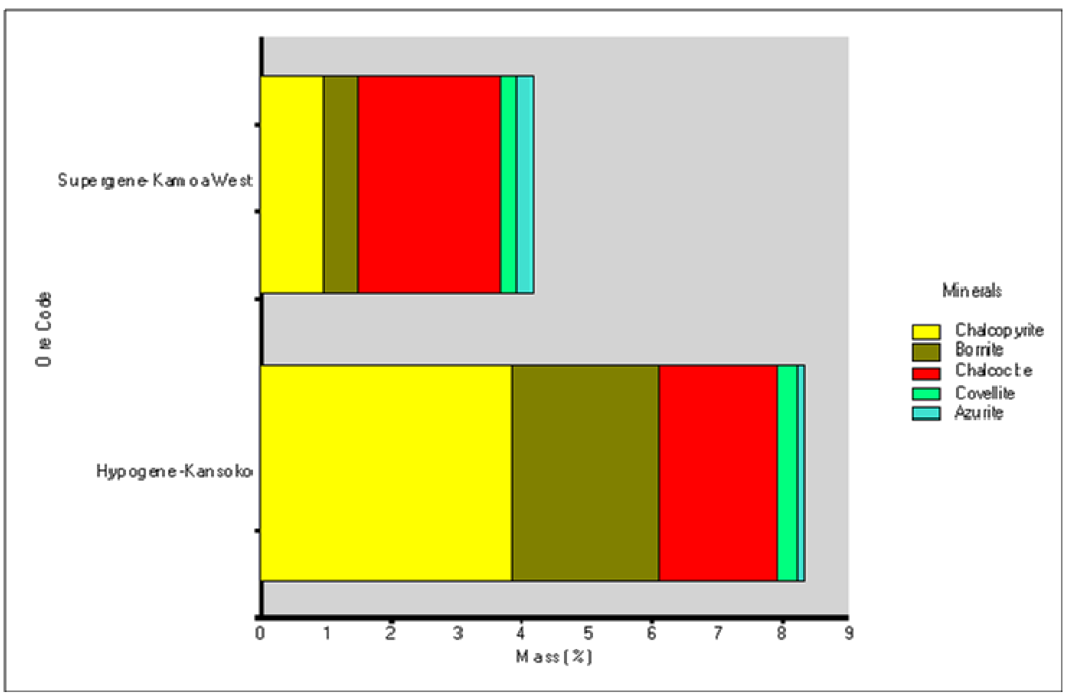
Figure 13-2 summarises the bulk modal mineralogy for the hypogene and supergene composites, and Figure 13-3 expands on the copper mineralogy components of each.

Figure 13-2: Modal Mineralogy of Hypogene and Supergene Feed Composites



Note: Figure courtesy XPS 2011

Figure 13-3: Cu Mineralogy of Hypogene and Supergene Feed Composites

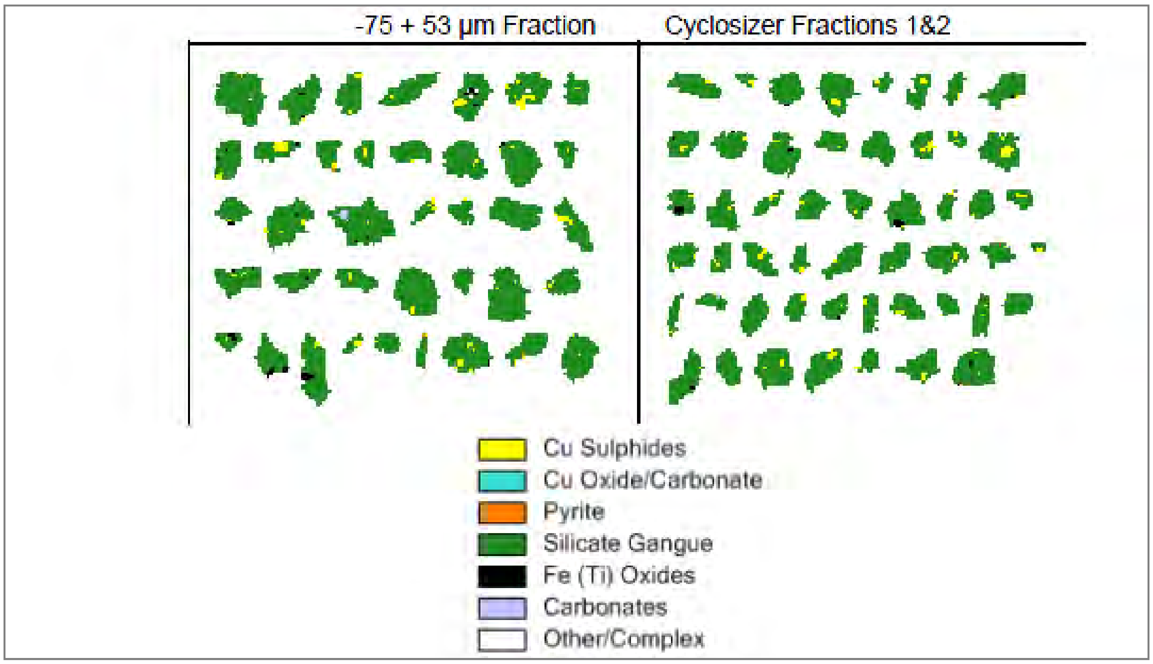


Note: Figure courtesy XPS 2011

The main gangue minerals in both samples are quartz, orthoclase, muscovite and chlorite. The hypogene copper mineralization is dominated by chalcopyrite and bornite followed by chalcocite. Supergene mineralization has proportionally more chalcocite with some chalcopyrite and bornite. Both samples contain small amounts of covellite and azurite.

A proportion of the copper sulphide mineralization has proved to be unrecoverable in flotation and the reasons were explored mineralogically. The main reason for recovery loss is finely locked hypogene copper minerals as demonstrated in the QEMSCAN image included as Figure 13-4.

Figure 13-4: QEMSCAN Image of Locked Copper in Flotation Scavenger Tail



Note: Figure courtesy XPS 2011

It is clear that the unrecovered copper (yellow) in the scavenger tails is locked, and present in fine phases (typically <10 µm). To recover the majority of the copper in the scavenger tails would require a costly (and probably unprecedented) whole-of-material grind to finer than 15 µm. The copper lost to the scavenger tail in this test is about 13%, and the majority of this can be classed as unfloatable.

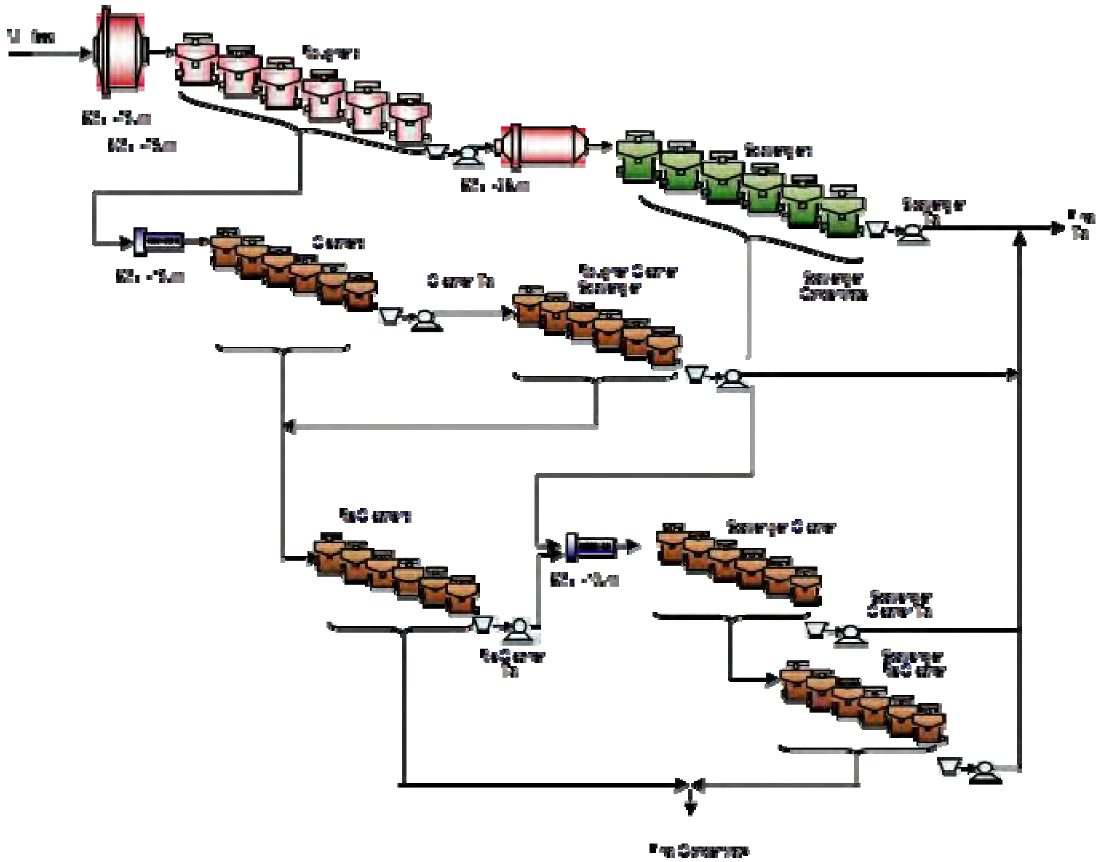
The initial supergene sample provided to XPS, and described in the analysis above, had a grade of only 1.9% Cu, and was representative of shallow potential open-pit material. The grade of this sample was significantly lower than the grade expected for the underground supergene resource which is above 3% Cu. Consequently a second supergene sample was prepared from known supergene intercepts in the region of the proposed underground mine, and this was then subjected to flotation testing. The head assay of this sample was 3.9% Cu. The comments on the supergene mineralization in this section refer to the original, low-grade sample. Information on the second supergene sample is given in the next sub-section and is discussed in Section 13.1.7.

Flotation Results

Flowsheet development at XPS commenced with the final Mintek MF2 flowsheet. A number of initial developments on this flowsheet were made, but the outcomes were similar to the final Mintek work, with copper recoveries at about 83% and concentrate grades at 27–28% Cu. The similarity of results is significant, because the comparison is being made between a shallow-sourced typical hypogene composite and a new, more representative, hypogene composite sample from much deeper mineralization. Some promise of higher concentrate grade was shown in these tests through the use of niche copper collectors.

A review of the results from initial XPS cleaning tests resulted in a partially optimised flowsheet solution that was nicknamed the “Sandton” circuit (after the review meeting location). The Sandton circuit achieved an improved outcome of 84% recovery at 28% Cu in concentrate. It achieved this through improving the recovery of copper from rougher concentrate to final concentrate from 96% to 98% without losing grade in the process. The concentrate grade achieved was relatively low, given the bornitic mineralogy, due to the presence of silica. Modifications to the Sandton circuit were tested and a circuit referred to as “Sandton Rev 3” achieved the best composite hypogene results in the program. The Sandton Rev 3 circuit is shown in Figure 13-5, and the results of the test are provided in Table 13-4. The Sandton Rev 3 results and flowsheet form the basis for the 2012 PEA.

Figure 13-5: Recommended Changes to the Sandton Circuit



Note: Figure courtesy Ivanplats, sourced from XPS, 2011

Table 13-4: Sandton Rev 3 Circuit Flotation Summary

Stream	Wt%	% Cu	% SiO2	Recovery Cu (%)
Final Concentrate	**8.6**	**32.8**	**19.1**	**85.4**
Final Tail	91.4	0.53	65.1	14.6
Feed	100	3.3	60.9	100

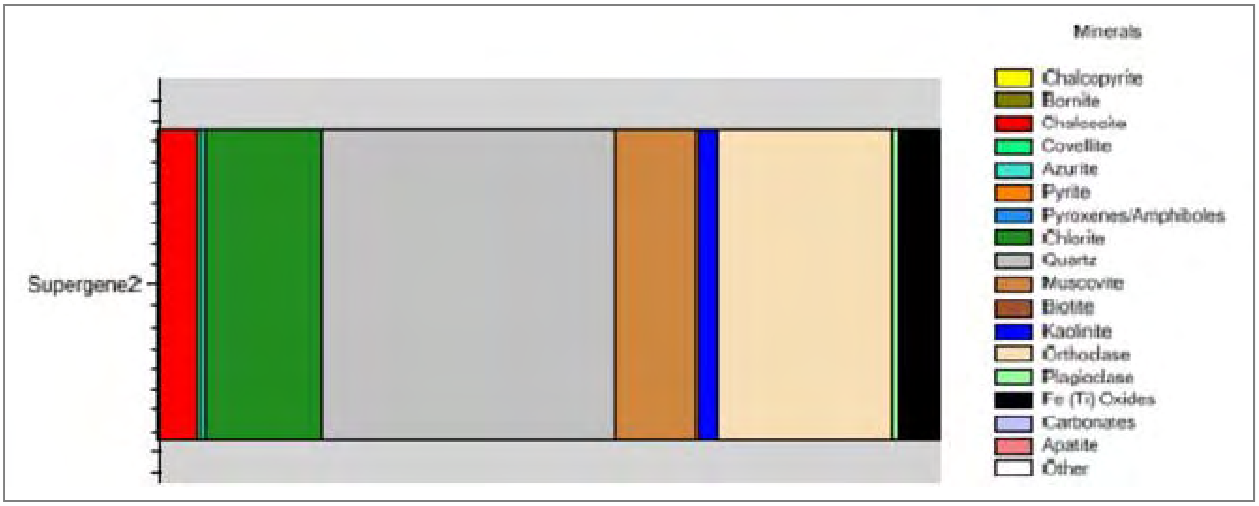
Preliminary Supergene Results – New Sample

The initial supergene sample provided to XPS had a grade of only 1.9% Cu and was representative of shallow open pit material. The grade of this sample is significantly lower than the grade expected for the underground supergene resource which is above 3% Cu. Consequently a second supergene sample was prepared from known supergene intercepts in the region of the proposed underground mine and this was then subjected to flotation testing. The head assay of this sample was 3.9% Cu.

Mineralogy

QEMSCAN analysis results are summarised in Figure 13-6 and Figure 13-7.

Figure 13-6: Modal Mineralogy Characterized of Supergene Material (after XPS, 2012)



Note: Figure courtesy XPS 2012

Figure 13-7: Elemental Deportment Element Mass Cu (after XPS, 2012)

Note: Figure courtesy XPS 2012

The main features to note are:

- Cu deportment is dominated by chalcocite (carries 79% of total Cu)
- Only 2.4% of total Cu is carried within Cu–Fe sulphides (mainly bornite) and the remainder is carried within covellite (7.7%) and azurite (9.3%).

Flotation Results

A preliminary test has been conducted on this high-grade supergene composite using the Sandton Rev 3 circuit developed for hypogene. The results are summarised in Table 3-5.

Table 13-5: Supergene Composite Flotation Results – Preliminary Test

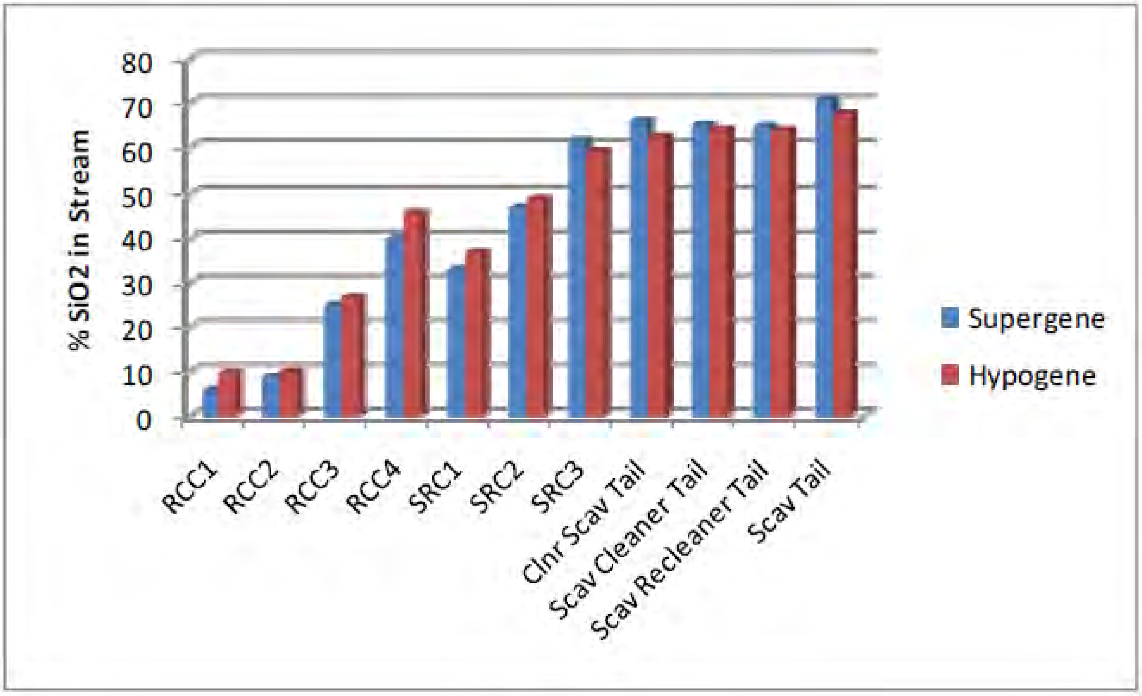
	High Grade Supergene Comp.									
	Grades					Recoveries				
	% Cu	**% S**	**% Fe**	**% As**	**% SiO2**	**Cu**	**S**	**Fe**	**As**	**SiO_2**
Final Concentrate	45.1	10.7	5.02	0.003	26.0	83.2	86.3	6.46	7.08	2.69
Head Grade	3.89	0.86	5.20	0.002	67.2	100	100	100	100	100

This result is confirmation that the flotation scheme developed for hypogene mineralization has worked without problem on a supergene sample. The concentrate grades were very high, with the first two rougher concentrates (representing almost 60% Cu recovery) assaying more than 68% Cu.

For a number of reasons, this result was not used as the basis for resource supergene performance in the 2012 PEA report. These reasons are:

- The result is from the only supergene test performed by XPS at the time of the report
- The test was done after calculations and estimations for the report were completed
- The concentrate grade achievable from supergene samples is dependent upon copper mineralogy. As supergene mineralogy is always variable, a program of work is in place to establish the relationship between the test sample and the average supergene resource copper mineralogy. Only when this relationship is established can a basis for concentrate grade prediction for supergene material be developed. The wide variation in copper content of the main copper minerals in the supergene mineralization, chalcocite (80% Cu), bornite (63% Cu) and chalcopyrite (35% Cu), is the reason that mineralogy is essential
- The supergene results provide excellent commonality with the hypogene results in terms of gangue response as, product for product in the flotation tests, the silica grades are almost identical (Figure 13-8). As silica is the major concentrate contaminant, this behaviour will probably form the basis for concentrate grade prediction once the average supergene copper mineralogy is known
- Consequently, the grade and recovery has not been used in the model.

Figure 13-8: Silica Grades in Flotation Test Concentrates with Flowsheet Sandton Rev 3



Note: Figure generated by AMEC, sourced from XPS, 2011

13.1.6 Conclusions Pertinent to the 2012 PEA

XPS development work on the hypogene cleaner circuit has advanced the circuit layout and has improved the yield of copper. Important outcomes are:

- Separate treatment of rougher and scavenger concentrates allows separately optimised cleaner flotation of fast and slow species.
- The IsaMilling of rougher concentrate to a P80 size of 15 µm allowed the recovery of 76% of the copper into a concentrate grading 37% Cu.
- The IsaMilling of combined middlings streams to a P80 size of 10 µm allowed the recovery of a further 9.7% of the copper at a scavenger re-cleaner concentrate grade of 17% Cu.
- Overall copper recovery was 85.4% at a grade of 32.8% Cu.
- Addition points for the primary collectors SIBX and 3477 have been identified.
- The addition of the Cytec niche collectors Aero 3894 (25 g/t) and Aero 5100 (5 g/t) to the IsaMill feeds and the IsaMill products respectively has improved the grade of the products from the rougher and scavenger re-cleaner banks.

- Re-cleaner concentrate grades in excess of 40% Cu have been achieved but only at low recoveries.

Mineralogical characterisation of the hypogene scavenger tailings identified and quantified the nature of the 11.7% of copper recovery loss to this stream. The losses are dominated by incomplete liberation associated with very fine copper sulphide grain size:

- Approximately 47% of all copper lost to scavenger tails is hosted as bornite, followed by 23% as chalcocite.
- The majority (59%) of all copper sulphides present are locked, 25% are middling, and a maximum of 16% are liberated in the raw two dimensional analysis. Typical conversion from two dimensions to real liberation in three dimensions will result in a reduction in the proportion liberated and corresponding increases in the locked and middling classes.
- Bornite, which forms the major copper loss, follows the above-described general liberation pattern. In the middling liberation class, the grain size averages 10 µm. The locked liberation class grain size averages 7 µm.
- Longer scavenger flotation times and a formulation of mixed collectors has resulted in improved flotation of the liberated and middling particles into scavenger concentrates.
- IsaMilling down to 10 µm has been necessary to achieve acceptable concentrate grades from middling streams confirming the liberation issues identified above.

13.1.7 XPS Testwork (August 2012 to March 2013)

New composites of hypogene and supergene mineralization were prepared by XPS for this test program. One unusual aspect of the new supergene sample was the low natural pH of 4.6 (compared to pH 8.0 to 8.5 for the Phase II supergene sample) and this appears to have had a negative influence on its flotation performance.

Testwork conducted at XPS since August 2012 has looked at a number of issues including:

- Changing from ball mill grinding to rod mill grinding
- Changing the regrind size distribution P80 targets
- Conducting primary and secondary rougher flotation under alkaline conditions rather than natural acidic conditions

- Conducting locked cycle testing with a modified circuit.

While the testwork has resulted in some encouraging performance improvements, the program is ongoing and no firm conclusions can yet be drawn. Copper recovery improvements of between 2% and 5% have been noted, partially as a result of using rod milling rather than ball milling, and partially as a result of using alkaline flotation conditions. However, additional testwork is required before it is possible to modify the existing recovery values and equations detailed in Section 13.2.

13.1.8 Comminution Testwork

Sample Selection

Mineralization at Kamoa occurs in two distinct lithologies, a siltstone and a diamictite (refer to Section 7). The siltstone is the mineralized downward extension of the hanging-wall pyritic siltstone, although a second “intermediate” siltstone also occurs in some areas. The diamictite occurs below the siltstone and grades from a finer-grained clast-poor variety into a coarser clast-rich variety. The former generally contains the greater part of the mineralization.

For an initial assessment of comminution performance, separate tests were performed upon samples of the upper siltstone, the clast-poor diamictite and the clast-rich diamictite. Owing to the relatively thin nature of the mineralized zone it was not possible to obtain adequate quantities of sample as continuous runs of drill core, and so “composite” samples were collected of quartered NQ core having a generally consistent appearance but derived from different boreholes in the initial target area.

A second set of comminution tests were carried out on eight new composite samples representing various horizons from the hanging wall through to the footwall.

Bond Work Index and Bond Rod Work Index Tests

Standard Bond rod-mill work index (BRWI) and ball-mill work index (BBWI) tests were conducted on the samples after crushing to -13.2 mm and -3.35 mm respectively, as per the standard procedure.

The BBWI values at a 106 um closing screen for all the samples from the diamictite mineralized zones were in the range 13 to 17 kWh/tonne. These results confirm that the mineralized materials are moderately hard, and harder than the typical Copperbelt ores. Given this deposit is not typical of Copperbelt shale-hosted mineralization, these results are not unexpected.

The BBWI results (at 106 um screen) for the hanging wall material went as high as 20 kWh/t while the footwall (including the underlying sandstone) was similar to the upper values for the mineralized zone of 16 to 17 kWh/t.

At a 75 um closing screen the BBWI values decreased by 0.5 to 0.8 kWh/t (compared to the 106 um closing screen results) for the three samples tested. This is an unexpected result because a finer grind usually results in less efficient grinding and a higher BBWI.

The implications of the BBWI results are that the materials are moderately hard, but do not get harder with finer grinding. It is also clear that the hanging wall material, the most likely source of dilution during mining, is predicted to cause grinding circuit throughput issues if it becomes an excessive component in the plant feed stream.

The BRWI values for all samples tested were in the range 17 to 24 kWh/t. These results show mineralization that is harder to grind at coarser sizes than finer, a result that is consistent with the high measured competence values discussed in the following section. Only the pyritic siltstone had RWI values of 20 and above, again indicating that hanging wall dilution with pyritic siltstone will cause plant throughput issues. These results class the material as hard to very hard with respect to rod milling. Footwall results were, again close to the upper results for the main mineralized types.

A comprehensive program of variability testing is essential to arrive at a definitive milling circuit design and is recommended before the definitive study phase commences.

SMC Tests

SMC (SAG mill competence) tests are primarily used to measure the competence of mineralized materials, and the results are relevant to semi-autogenous grind (SAG), autogenous grind (AG) and ball milling as well as high-pressure grind roll (HPGR) design. The test is based around the same principles as the JK Drop Weight test but is carried out on a single particle size rather than five sizes. The results are summarised in Table 13-6.

Table 13-6: Mineralized Material Competence Results

Sample Designation	A * b Value	Category	Rank	%
Lower Pyritic Siltstone	29.4	Very Hard	371	12
Clast_Poor Diamictite	38.0	Moderately hard	975	32
Clast_Rich Diamictite	36.8	Hard	884	29
Diamictite Mineralized (Comp 1)	31.0	Hard	571	16
Diamictite Mineralized (Comp 2)	22.2	Extreme Hard	96	2.6
Diamictite Mineralized (Comp 3)	23.9	Extreme Hard	140	3.9
Pyritic Siltstone Mineralized (Comp 4)	21.5	Extreme Hard	73	2.0
Pyritic Siltstone Hanging Wall (Comp 5)	21.2	Extreme Hard	71	2.0
Diamictite Hanging Wall (Comp 6)	28.3	Very Hard	374	10
Sandstone Foot Wall (Comp 7)	24.7	Extreme Hard	166	4.6
Diamictite Foot Wall (Comp 8)	23.3	Extreme Hard	128	3.5

The Kamoa samples all have high to extreme competence. SAG milling these materials would be inefficient and is not recommended at any scale of throughput being contemplated without the use of fine secondary crushing.

Abrasion Testing

Only one abrasiveness result was available for the 2012 PEA and this was for clast-rich diamictite with a value of 0.139. This is characteristic of a low abrasiveness material.

Apart from the footwall samples, the recent Bond Abrasion Index (Ai) results have confirmed the low abrasiveness status of the material. The hanging wall and mineralized zone results range from values of 0.036 up to 0.11. The footwall diamictite (0.18) has moderate to low abrasiveness and the footwall sandstone is worst of all at 0.378. Dilution of the plant feed stream with footwall sandstone should be avoided.

The new results suggest that the previous Ai design value of 0.139 is conservative and future studies should use a lower value. The implications of lower abrasiveness are lower consumption of grinding media and slower wear rates on mill and crusher liners.

Comminution Discussion

The tested Kamoa mineralized materials display a very high degree of competence and high grindability indices. The choices for milling mineralization of this type include single-stage semi-autogenous grind (SAG) mills, secondary crush SAG milling, semi autogenous-ball mill-crushing (SABC) circuits, high-pressure grind roll (HPGR) circuits and conventional cone crush and ball mill circuits. For smaller throughputs, rod and ball mill circuits can also be considered. A disadvantage of a SAG-style circuit is that the throughput will always be much more variable than with a crush and ball mill circuit. Variable throughput results in flotation inefficiencies which may be tolerable with a low-grade high-throughput circuit but are not desirable when treating high-grade feeds in complex flotation and regrind circuits.

At the throughput rates envisaged in the 2012 PEA, a multiple-stage crush circuit followed by ball milling was recommended. Consideration could be given to the third crushing stage being an HPGR, but given the Project location, the simplicity of cone crushing is likely to provide a more robust solution.

The target throughput rate has increased in the current studies (after August 2012), and the comminution circuit choice will be re-evaluated in light of this change.

13.2 Recovery Estimates

13.2.1 2012 PEA Recovery Assumptions

Hypogene

From the testwork results a recovery algorithm was developed which was used to calculate copper recovery into a constant concentrate grade with varying head grade. The algorithm considered a reference feed head grade of 3.3% Cu and a copper recovery of 85.4%. The concentrate grade is constant at 32.8% Cu.

The following algorithm was developed:

Copper Recovery % = [%Cucon * (%CuFd - %Cutail)]/[%CuFd * (%Cucon - %Cutail)]

Where:

%Cutail = %Cunf + [(%Cutref - %Cunf)/%Cufdref * %Cufd]

%Cucon = %Cu in concentrate, 32.8%

%CuFd = %Cu in feed

%Cunf = %Cu non floating, 0.3%

%Cutail = %Cu in tailings

%Cutref = %Cu reference tailings grade, 0.53%

Supergene

Due to the absence of an optimising test program the hypogene algorithm was used for supergene. The algorithm is referenced to a feed grade of 1.88% Cu and a copper recovery of 70%. The concentrate grade is constant at 29.3% Cu. The following algorithm was developed:

Copper Recovery % = [%Cucon * (%CuFd - %Cutail)]/[%CuFd * (%Cucon - %Cutail)]

Where:

%Cutail = %Cunf + [(%Cutref - %Cunf)/%Cufdref * %Cufd]

%Cucon = %Cu in concentrate, 29.3%

%CuFd = %Cu in feed

%Cunf = %Cu non floating, 0.3%

%Cutail = %Cu in tailings

%Cutref = %Cu reference tailings grade, 0.59%.

AMEC notes that work at XPS has been initiated on the supergene mineralization, but not finalised. When supergene work is complete a set of updated algorithms will be developed, which are likely to vary from the algorithms presented above.

13.3 Metallurgical Variability

A program of metallurgical variability testing has not yet been conducted but samples have been identified for this work. The variability testing will look at a number of examples of each mineralized type and will also test full mineralized zone composites from a number of locations around the deposit. The full mineralized zone composite basis is the most important variability test unit, as the entire zone will be mined at the one time in each location.

The planned program is suited to the current PEA update study and proposed future pre-feasibility study phase.

13.4 Deleterious Elements

Testwork to date has shown no penalty elements present to problematic levels in the concentrate. The main impurity element is silica, and testwork is being undertaken to minimise silica recovery.

Saleable concentrate generated from the XPS test program in February 2012 averaged 32.5% Cu and contained 80 ppm As, 1.1% MgO, 5.0% Al_2O_3 and 19.0% SiO_2.

13.5 Comments on Section 13

The AMEC QPs made the following comments on the testwork available to support the 2012 PEA:

- The metallurgical test program to date has been successful in generating saleable and smeltable copper concentrates and has provided the basis for comminution circuit design.
- The flotation work has shown a consistency of outcome strongly driven by the consistent liberation characteristics of the copper mineralization. All indications are that the maximum recoverable copper to concentrate with the dominant hypogene mineralization will be in the region of 85% at a final concentrate grade in the 32 to 35% Cu range. Results from preliminary tests on the supergene mineralization suggest that concentrates of 40% copper, or more, may be achievable at ultimate recovery. Representative test work results are expected from XPS later in 2012.
- Further flotation testwork is required to establish the variability of the mineralized material response and provide guidance as to the blending needs for concentrator feed control. The XPS flotation program is ongoing and will be addressing supergene circuit optimisation, flotation of oxide mineralization and hypogene variability testing.
- Comminution test work has led to a crush and ball milling circuit being the preferred method of preparing flotation feed.

The AMEC QPs have based the following comments on additional testwork in progress since the completion of the 2012 PEA. As a result of this work, there is assumed to be no impact on the proposed plant performance assumptions, or the capital and operating cost estimates in the 2012 PEA. The additional testwork results have provided the following specific assurances:

- The sample competence is higher than was previously assumed, but this only strengthens the case for preferring a crush and ball mill circuit over a SAG mill based circuit.
- Higher competence may result in additional crushing and screening requirements but crushing design is typically based on Bond Crusher Work index results (CWI), none of which are currently available. Extraction of suitable diameter core for CWI measurement is recommended before a definitive crusher design is possible.
- The design grindability values (BWI and RWI) have not changed significantly with the new results. The most serious implication from the new results is the negative effect that the harder pyritic siltstone values will have on mill throughput. In mining locations where pyritic siltstone is the main mineralized material, then its proportion in mill feed will need to be monitored and controlled.
- The Bond abrasion index used in design is conservative
- The copper recovery from hypogene material will probably prove conservative once new test conditions are established
- The copper recovery and concentrate grade from supergene mineralized samples continues to be variable, and a program of variability testing is needed to define the nature of supergene across the resource.

14.0 MINERAL RESOURCE ESTIMATES

14.1 Key Assumptions/Basis of Estimate

The underground and open pit resource models discussed in this section were constructed by Gordon Seibel, R.M. SME, of AMEC.

The AMEC QPs consider that the Mineral Resource models and Mineral Resource estimates derived from those models are consistent with Canadian Institute of Mining Metallurgy and Petroleum (CIM) Definition Standards (2010) and the relevant CIM Best Practice Guidelines (2003).

14.2 Composites for Mineral Resource Estimates Amenable to Underground Mining Methods

Mineral Resource estimates used 543 drill hole intercepts (including six twins) over an area of approximately 80 km^2 (Domain 1) where the drill hole spacing is close enough (100 m to 800 m) to allow mineralization to be classified as Indicated and Inferred (refer to Figure 10-2). The area outside of Inferred and Indicated Mineral Resources, but within the Project boundary (Domain 2), is approximately 63 km^2 in extent and was modeled to help estimate ranges of tonnages and grades of potential Exploration Targets using an additional 12 drill holes with drill hole spacings of up to several kilometres.

Collar, survey, lithology and assay files were exported from the Ivanplats' Access® database as csv files, imported into Datamine Studio® mining software, and combined into a "holes" file. The holes file was then exported to Excel®, and a SMZ selection field was added, and set to a default value of "0". The highest-grade single intercepts were then selected, and the SMZ selection field was flagged as "1" by hand if the following criteria were met:

- Minimum down hole length 3 m (locally adjusted to approximate 3 m vertical thickness for angled core holes)
- Total copper (Cu) grades greater than 1% Cu.

The 1% Cu cut-off was used as it is considered a natural cut-off, and is commonly used in similar deposits. The SMZ were first composited manually in Excel®. In the event that a hole did not meet the minimum grade ≥1% Cu and length greater than 3 m criteria, the highest-grade composite was formed using a length similar to adjacent-hole SMZ grading ≥1% Cu.

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 drill hole at different elevations. In these cases, only the highest-grade composite was used to introduce a degree of conservatism, as it has not yet been determined whether multiple SMZs can be extracted. Statistics for the SMZs are given in Table 14-1.

The “holes” file with the SMZ selection field was then imported into Datamine mining software, and Cu assays greater than 16% were set to 16%, and ASCu assays greater than their corresponding Cu assays, were set to the Cu grade. The cap or top-cut resulted in the average grade of the composites being reduced by approximately 1% (relative). The drill-hole file was then composited over the entire interval where the SMZ selection field equalled “1” in Datamine®, forming a single composite for each hole with variable length. The composite grades calculated in Datamine were then compared to the composite grades calculated by hand in Excel and found to be identical except for an insignificant amount of rounding error, or differences due to capping.

14.3 Exploratory Data Analysis

Exploratory data analysis (EDA) was performed on the SMZ composites to better understand the data used in the Mineral Resource estimate. The following techniques (primarily graphical) were used to reveal the underlying characteristics of the data:

- Summary of univariate statistics for major elements (Table 14-1)
- Univariate histograms and cumulative probability plots for all fields to display the summary statistics graphically (Figure 14-2 to Figure 14-5)
- Bivariate scatter plots to show the relationships between the different variables (Figure 14-6).

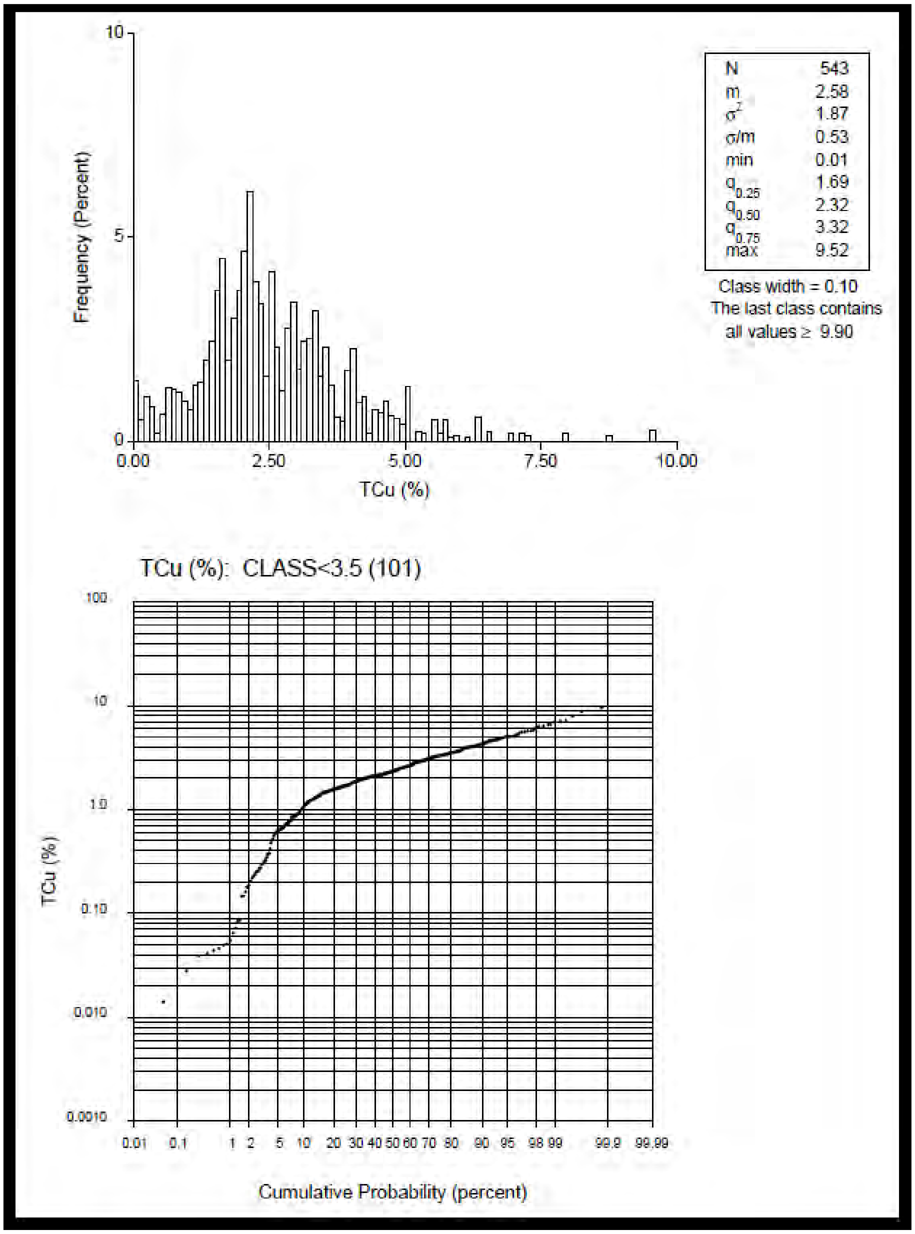
The following abbreviations were used in the table and figures:

- Cu = Total copper (TCu)
- ASCu = Acid-soluble copper
- RATIO= ASCu/Cu
- True Thickness (TrueThk) = True thickness, i.e. thickness perpendicular to the SMZ
- CV = Coefficient of variation = standard deviation ÷ mean

Table 14-1: SMZ Composite Statistics

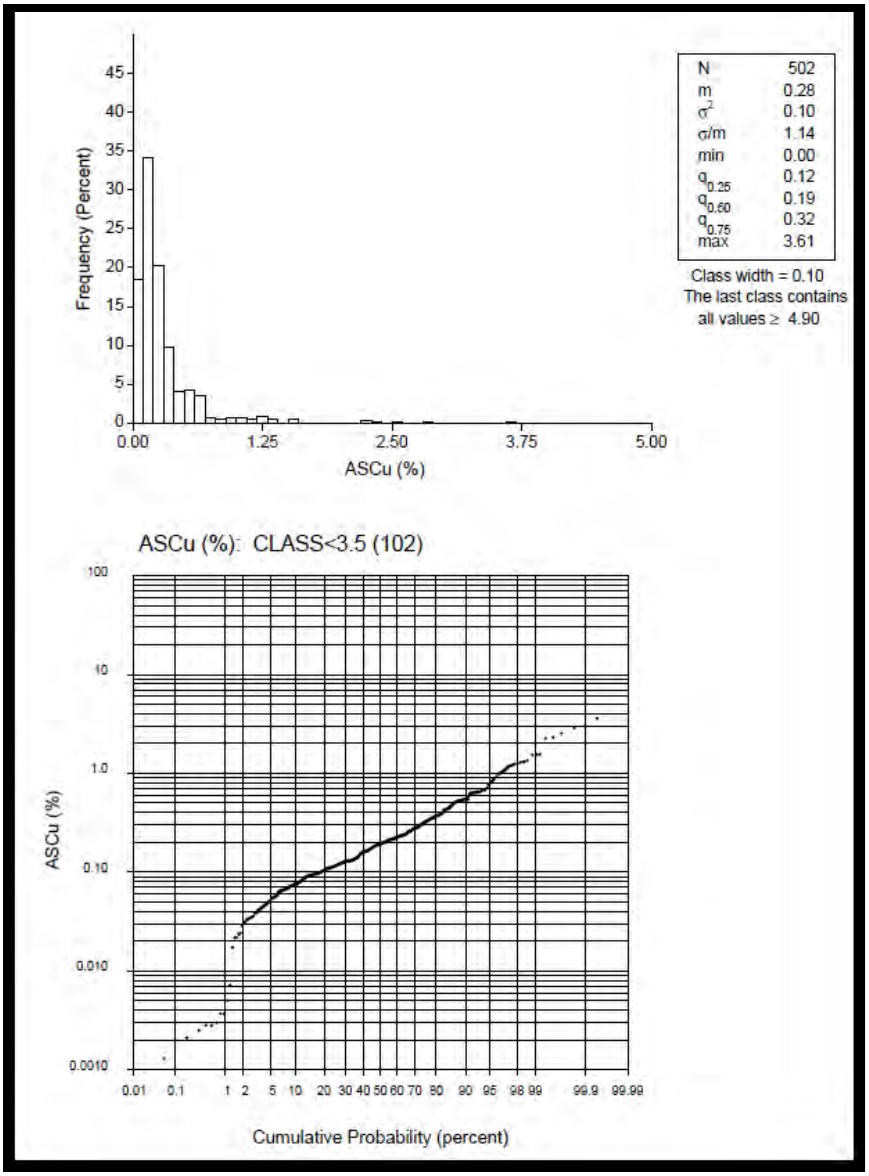
Variable/Area	Number	Minimum	Maximum	Mean	Standard Deviation	Coefficient of Variation
%Cu (capped)						
Domain 1	543	0.01	9.52	2.40	1.44	0.60
Domain 2	12	0.32	4.00	1.96	1.24	0.63
%ASCu						
Domain 1	502	0.00	3.61	0.28	0.32	1.14
Domain 2	12	0.00	0.00	0.00	—	—
True Thk (m)						
Domain 1	543	2.17	18.60	5.23	2.75	0.53
Domain 2	12	2.98	14.79	5.36	3.26	0.61
Cu x TThk (m-%)						
Domain 1	543	0.04	64.36	13.43	10.87	0.81
Domain 2	12	0.96	59.20	10.50	15.20	1.45

Figure 14-1: SMZ Composite Cu; Histogram and Cumulative Probability Plot Weighted by True Thickness for Domain 1 (based on assay data top-cut to 16% Cu before compositing)



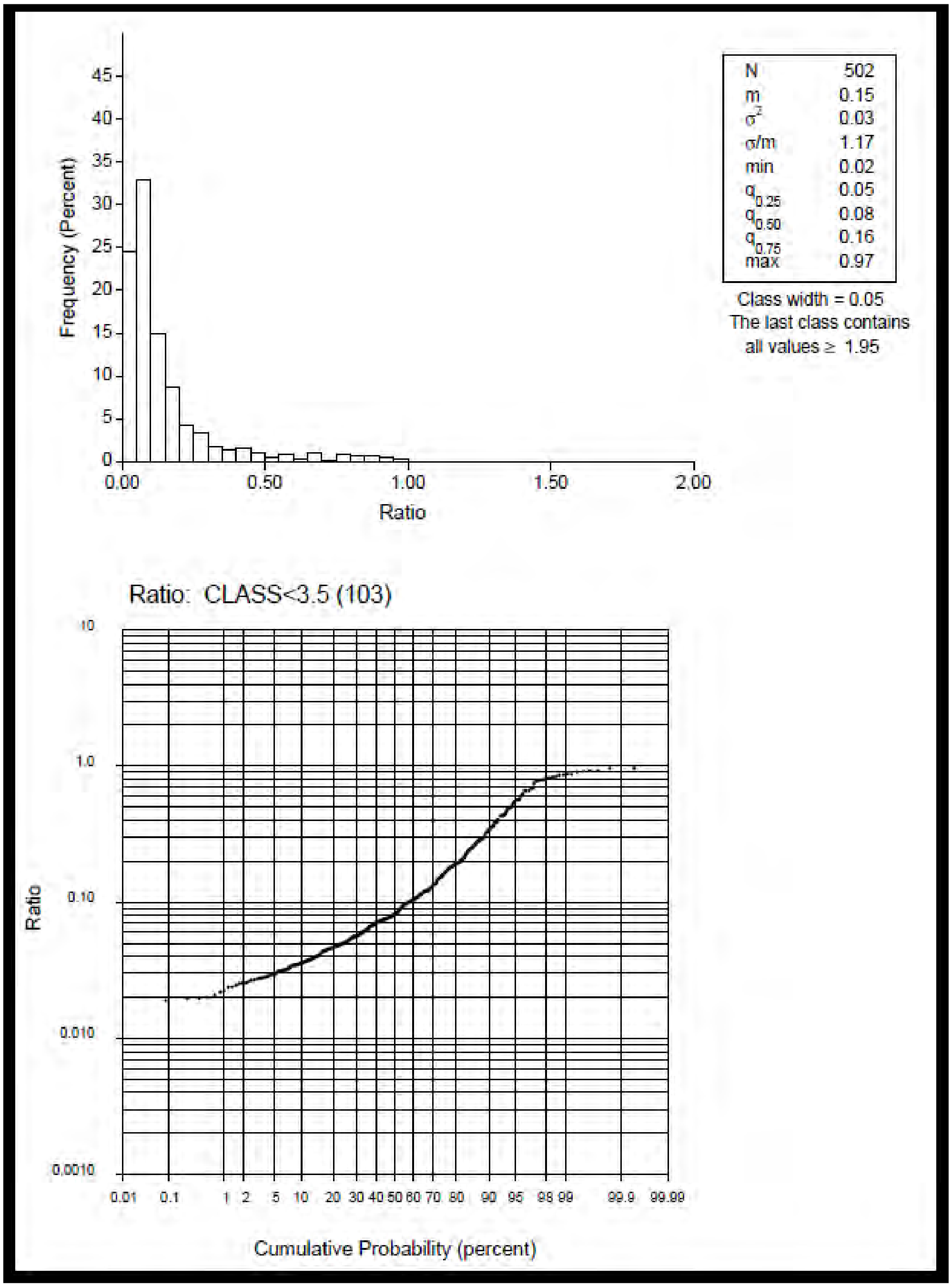
Note: Figure prepared by AMEC, 2013

Figure 14-2: SMZ Composite ASCu; Histogram and Cumulative Probability Plot Weighted by True Thickness for Domain 1 (ASCu set to Cu if ASCu>Cu)



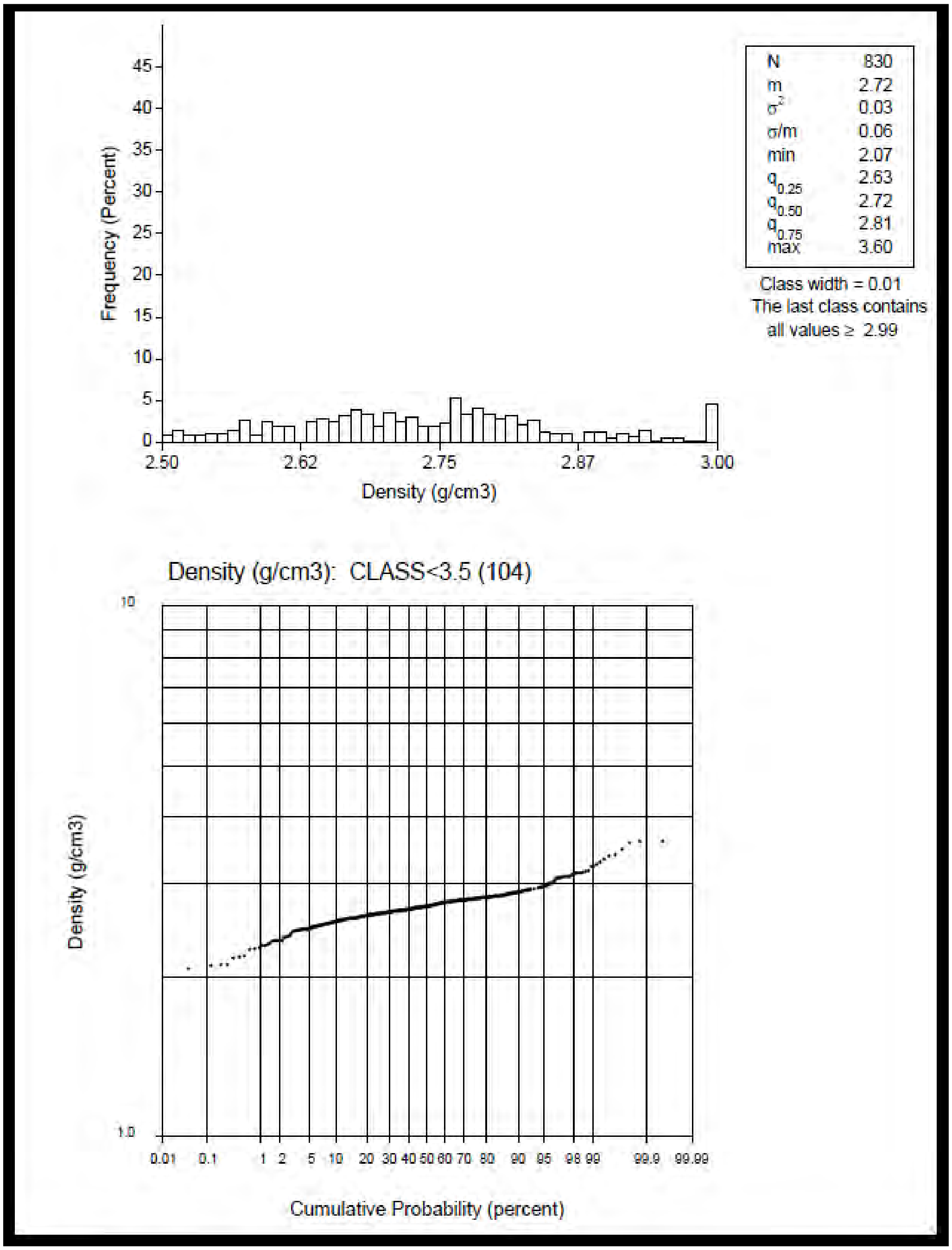
Note: Figure prepared by AMEC, 2013; figure based on 502 SMZ composites

Figure 14-3: RATIO (ASCu/Cu); Histogram and Cumulative Probability Plot Weighted by True Thickness for Domain 1



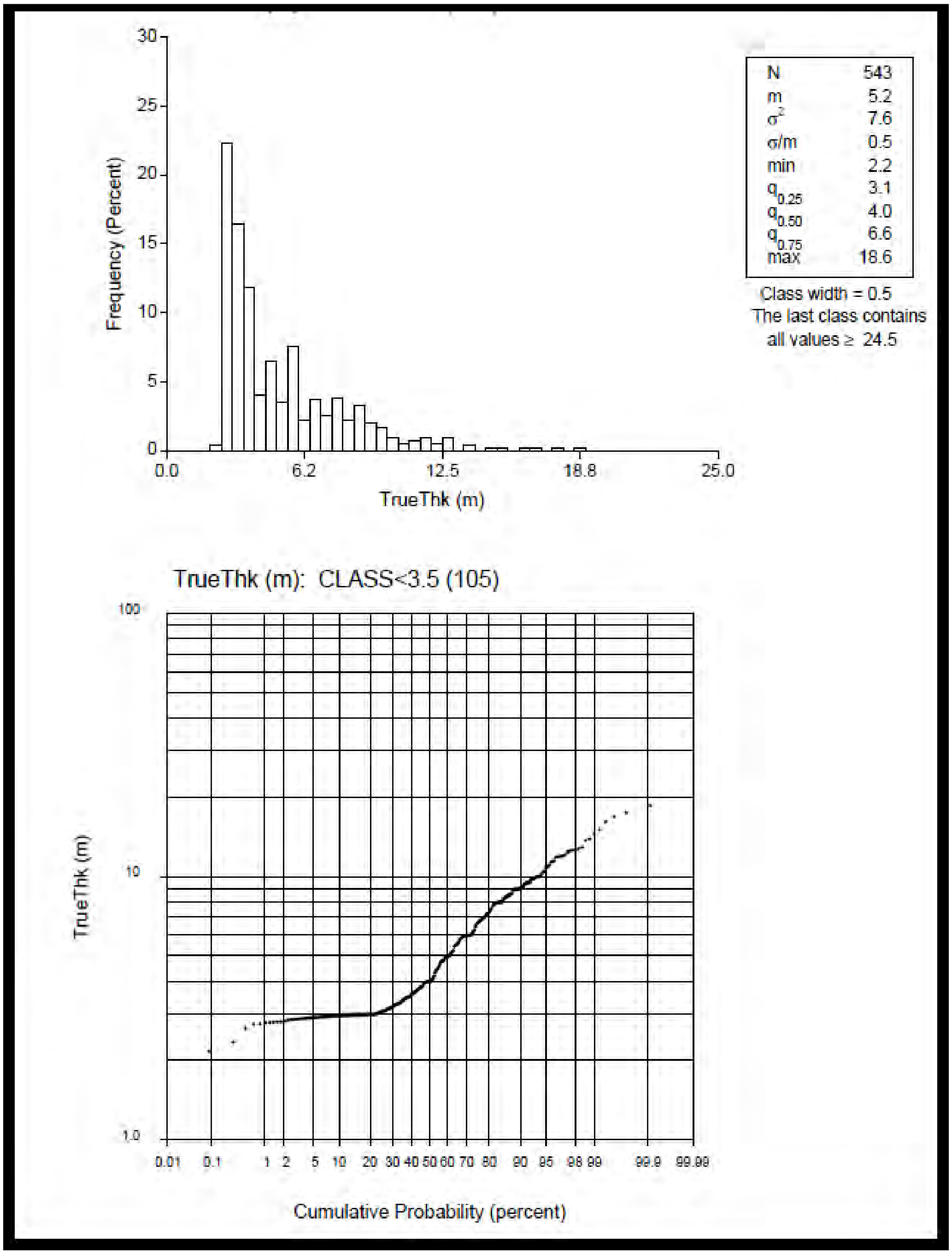
Note: Figure prepared by AMEC, 2013; figure based on 502 SMZ composites

Figure 14-4: Density Histogram and Cumulative Probability Plot for Density Assays for Domain 1



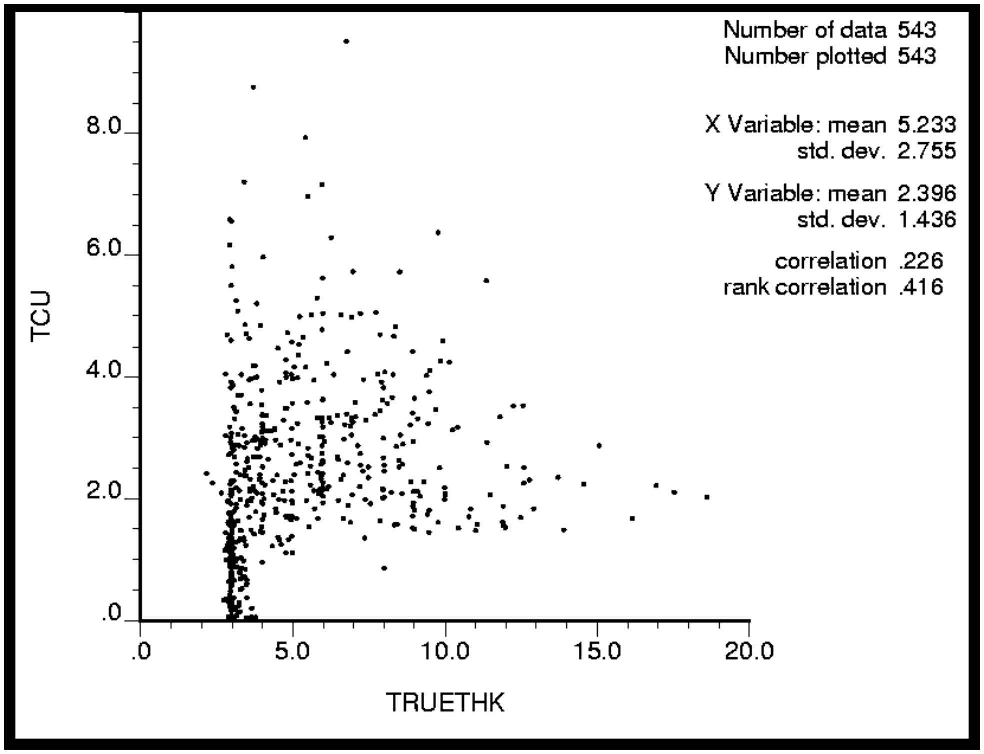
Note: Figure prepared by AMEC, 2013

Figure 14-5: True Thickness Histogram and Cumulative Probability Plot for SMZ Composites in Domain 1



Note: Figure prepared by AMEC, 2013

Figure 14-6: Scatter Plot of Total Copper (%) versus True Thickness (m)



Note: Figure prepared by AMEC, 2013

14.3.1 Univariate Statistics

Copper composites approximate a normal distribution, while ASCu and RATIO composites are positively skewed, tending towards a log-normal distribution.

Approximately 16% (91) of the Cu composite values are less than 1% Cu. Local internal zones of low grade or non-deposition of sulphides are characteristic of sediment-hosted copper deposits, and were included in the resource estimation to add lateral dilution. Many of these composites are situated around the edges of the drilled area.

A total of 41 composite values from Domain 1 are missing ASCu assays (and the resulting RATIO). Since the samples were never assayed for ASCu, the ASCu values were set to a null value (not zero) for estimation.

Approximately 80% of the composites with ASCu assays have a RATIO of less than 0.2 indicating that most of the deposit is sulphide; however, the remaining 20% of the composites with higher ratios suggest that there exist local areas of copper oxides or partially acid-soluble sulphides such as chalcocite.

The maximum RATIO value is 0.97 which is a result of adjusting the ASCu values to the corresponding Cu values if the ASCu assays were greater than Cu assays before compositing.

No major inflections were observed in the RATIO cumulative frequency plot, which if present, would suggest that separate and distinct copper oxide and sulphide populations may exist. However, a minor inflection does occur near the top 3% of the c indicating that small, localized oxide areas may occur, and domaining of a separate oxide domain should be considered when there are sufficient data.

The CV for density is very low (0.06), indicating that the density should be very consistent throughout the model.

14.3.2 Bivariate Statistics

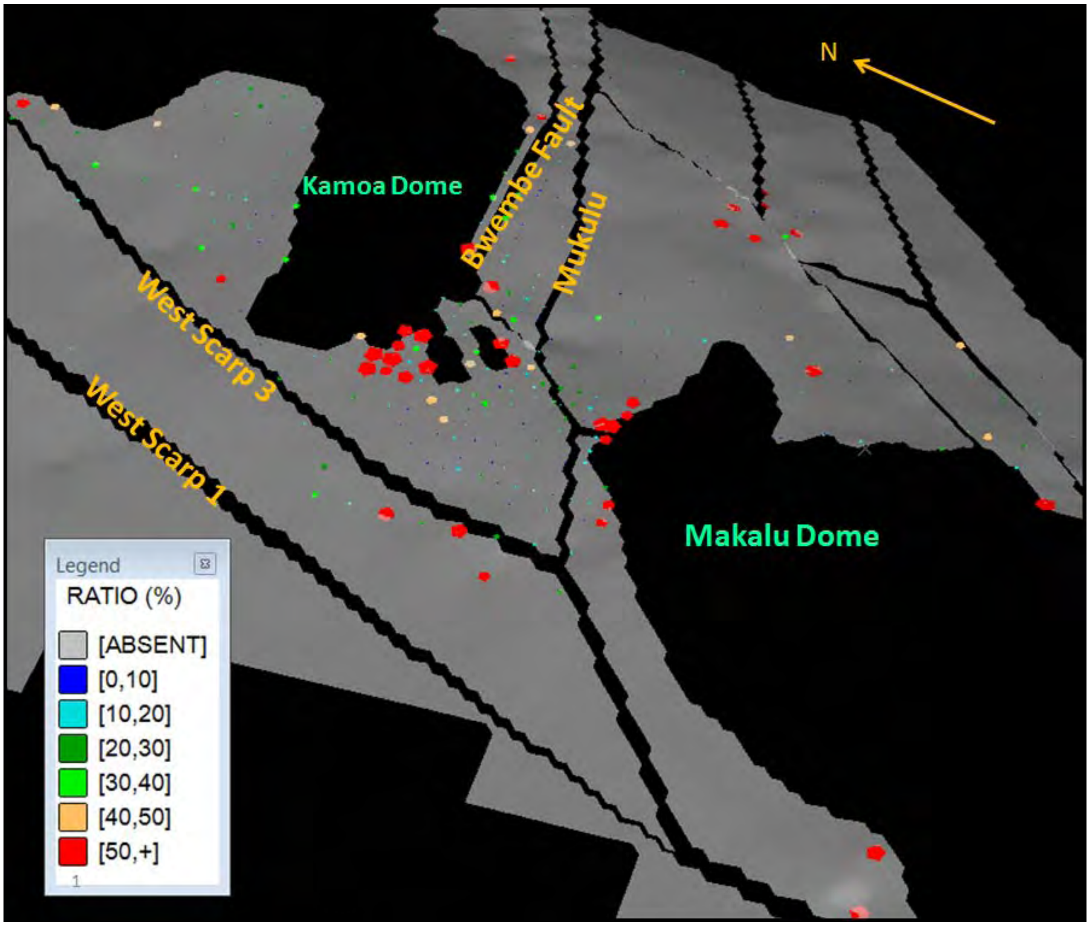
The relationships between the different variables were evaluated using scatter plots, as illustrated by the scatter plot for total copper against true thickness (refer to Figure 14-6).

The results presented in this figure indicate that although there is a very weak correlation between total copper and true thickness, the correlation is not statistically significant. Nevertheless, total copper estimations should be weighted by true thickness in case there are local areas with significant correlations. Similar plots were constructed for ASCu and the RATIO. No significant correlations were noted between acid soluble copper and true thickness, or the ratio of acid soluble: total copper to true thickness.

14.4 Oxide–Sulphide Domaining

Separate domains for oxides and sulphides were evaluated to preserve the spatial discontinuities of oxide and sulphide material. Initial reviews of the cumulative probability plot for the ASCu and the ASCu: Cu RATIO (refer to Figure 14-2 and Figure 14-3) did not show any major inflections to suggest two different populations, but a few composites with high ratios do exist. Review of the RATIO values spatially showed that the higher RATIO values occur along edges of the domes and along structures (Figure 14-7). From these studies, it was concluded that the small size of the oxide occurrences does not warrant a separate domain to constrain grade modelling at this time, but the high ratio areas may be useful for geometallurgical characterization and identifications of faults.

Figure 14-7: Perspective Schematic View of the ASCu:Cu Ratio



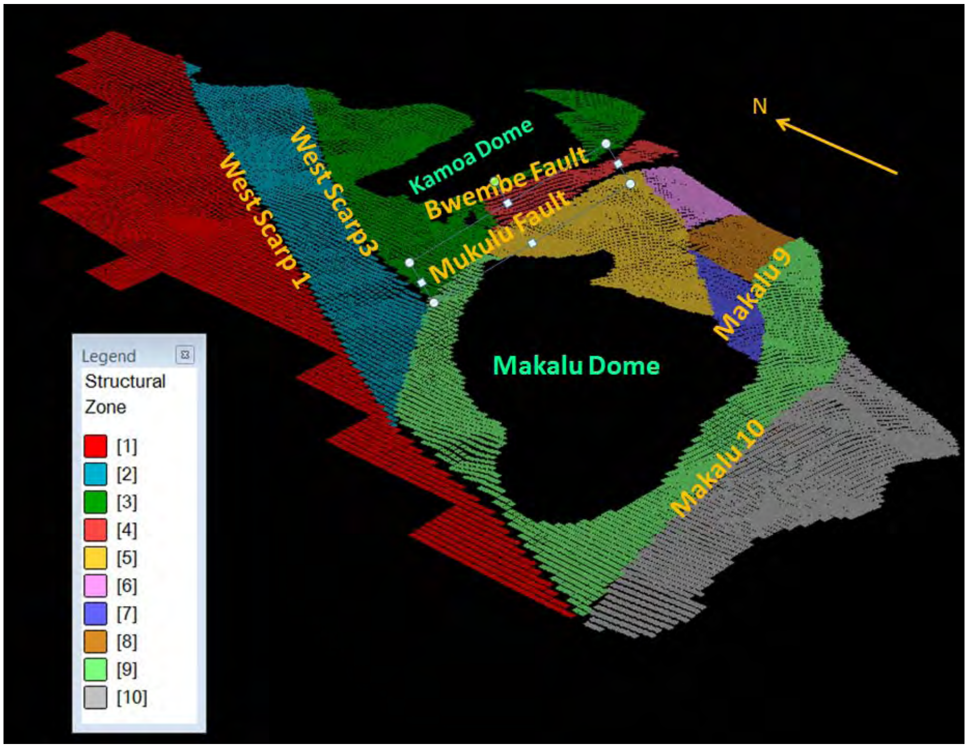
Note: Figure prepared by AMEC, 2012. Values in relation to the major faults (size of the points is proportional the magnitude of the ratio), no scale, looking northeast; colour-codes are based on copper percents as indicated in key.

14.5 Structural Model

Approximately 25 structures have been identified by SRK based on geophysical data, and lithological discontinuities interpreted from the drill hole data. For the resource estimate, these structures were spatially compared to the inflections in the geometry of the SMZ and nine were identified as offsetting the mineralization. These structures were then used as boundaries to divide the mineralization into structural zones where the mineralization inside each structural zone maintains a similar strike and dip.

The Kamoa resource model area was constructed by first evaluating the structural data (SRK, 2012), and dividing the deposit into 10 structural zones with similar strikes and dips based on geophysical data, and lithological discontinuities interpreted from the drill hole data (Figure 14-8).

Figure 14-8: Perspective Schematic View of Structural Zones that include the Faults and Domes used to define the Structural Zones



Note: Figure prepared by AMEC, 2012; looking northeast.

The zones are defined by the following major faults:

- West Scarp 1 fault (usually referred to simply as the West Scarp fault; strikes 20°, up to 300 m downward displacement on west side)
- West Scarp 3 fault (strikes 20°, up to 200 m downward displacement on west side)
- Bwembe fault (strikes 80°, 50 m to 400 m downward displacement on south side)
- Mukulu fault (strikes 80°, a scissors-fault with 0 m displacement on its western end to 400 m downward displacement of the south block on the eastern end of the fault)
- Makalu fault 9 east of the Makalu dome, (strikes 60°; 200 m downward displacement on south side)
- Makalu fault 10 fault (strikes 70°, a scissors-fault with variable upward or downward displacement.

There are inadequate data to determine the dip of these faults, and/or to determine if they are a single fault plane or a fault zone. For the Mineral Resource estimate, 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, but the available data are too wide-spaced to model them.

14.5.1 Copper Resource Model

The Kamoa underground copper resource model was constructed by Gordon Seibel of AMEC as follows:

- The cut-off date for exporting the drill holes from the database was 10 December 2012.
- The perimeter of the mineralization was defined using 555 mostly near-vertical, mineralized core drill holes that excluded barren leached drill holes where the mineralization approaches the surface.
- The best single mineralized intercept (SMZ) for each of the 555 holes within the resource boundary was selected using the criteria of a minimum copper grade of 1% Cu, and a minimum down-hole length of three metres. In the event that the assays in a drill hole could not be combined to meet the above criteria, the highest-grade composite was formed with a length similar to those of the adjacent SMZs.
- The mineral resource area was divided into 10 structural domains, and a digital terrain model (dtm) was constructed through the SMZ centroids to define the geometry of the mineralization for each structural domain.
- A prototype gridded-seam block model was established using 100-metre x 100-metre blocks in the X and Y directions and a single block in the vertical direction using the parameters in Table 14-2. The Z value of the block centroid was then set to the elevation of the SMZ dtm at the corresponding X and Y coordinate.
- A dtm wireframe was then constructed through the centroids of the SMZ blocks.
- Dip and dip direction were calculated for every center-of-gravity point in each triangle of the SMZ wireframe.
- The dip and dip-direction from the SMZ wireframe was then estimated into each block using by a moving window average that covered the adjacent blocks.
- The dip and dip-directions in the model blocks were then tagged to each composite that passed through that block, and used in conjunction with the length, azimuth, and inclination of the composites to calculate the true thickness of the mineralization for each composite.

Table 14-2: Block Model Parameters

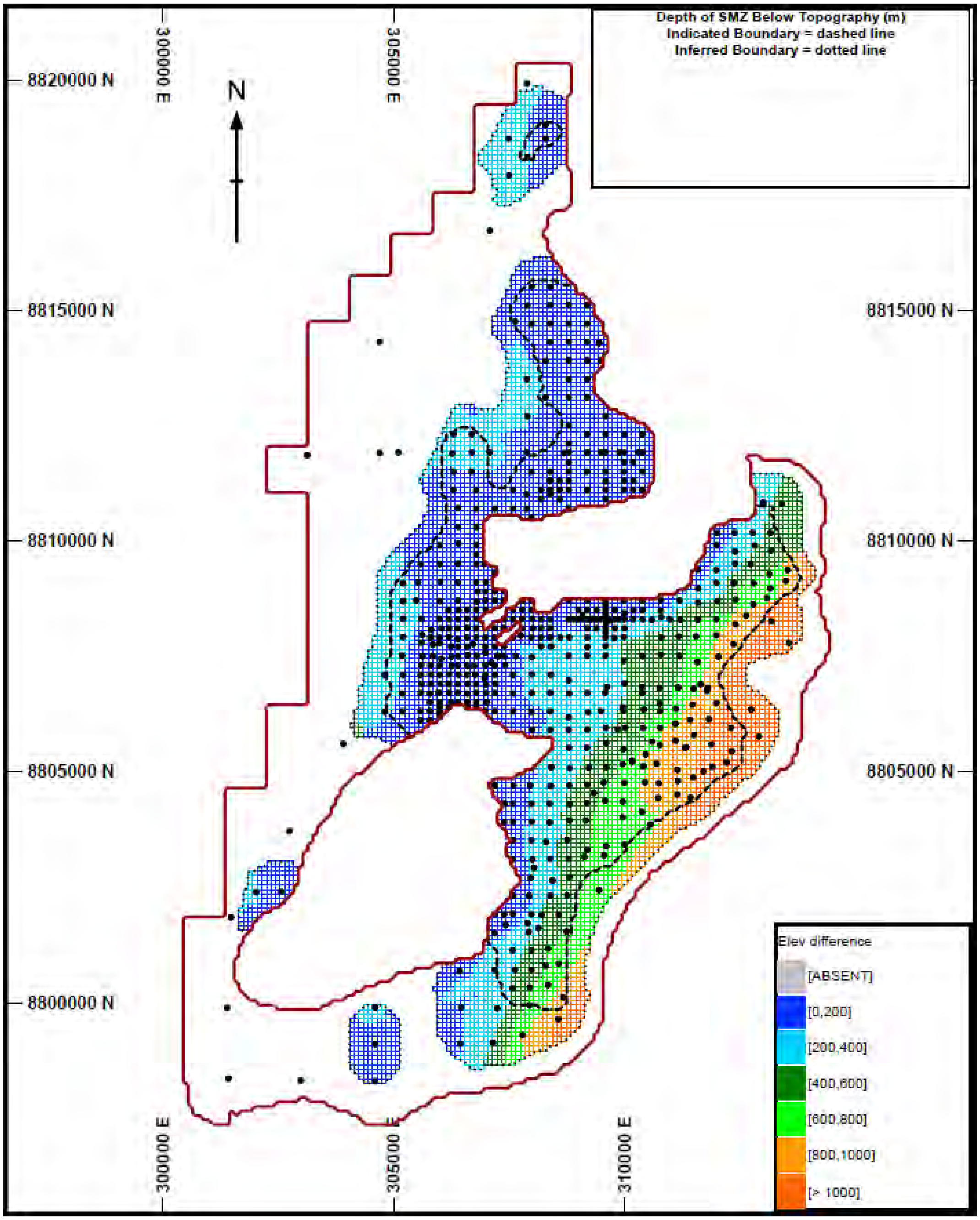
Axis	Origin	Maximum	Block Size (m)	# Blocks
Easting (X)	295,050	320,050	100	250
Northing (Y)	8,797,050	8,827,050	100	300
Elevation (Z)	N/A	N/A	Variable	1

- The calculated true thickness in the composites was then estimated into the model by an inverse distance to the second power (ID2) using search parameters shown in Table 14-3 (all subsequent estimates used ID2 and the search parameters listed in Table 14-3).
- Vertical height of the mineralization was calculated by dividing the true thickness by the cos of the dip, and the height of the block was set to estimated vertical thickness.
- Total copper (Cu) estimates were weighted by true thickness by first estimating Cu times true thickness and true thickness, and then estimating Cu by dividing Cu times true thickness by true thickness.
- Since acid soluble (ASCu) assays are not available for every Cu assay, ASCu was estimated by first estimating Cu and ASCu using only composites that contain both an ASCu and a Cu assay. From the paired data estimation, the ASCu:Cu ratio (RATIO) was estimated for each block and then the ASCu was estimated by multiplying the estimated Cu times the ASCu:Cu ratio derived from the paired data.
- Density was estimated into each block using only the density values within the mineralized horizon by selecting only the density values located between a wireframe constructed on the top SMZ blocks and the bottom of the SMZ blocks.
- Hanging wall and Footwall dilution skins 0.3 high were then constructed above and below the SMZ model using only those assays that lie within the 0.3 m above or below the SMZ block model.
- The Mineral Resources have been defined taking into account the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves. Resources were classified using the same criteria reported in the 5 September, 2012 Technical Report (Peters et al., 2012) that required a nominal 400 m drill hole spacing for Indicated and a nominal 800 m spacing for Inferred.
- The depth of the SMZ is shown in Figure 14-9.

Table 14-3: Estimation Parameters

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

Figure 14-9: Depth of SMZ (meters below surface; Domain 1)



Note: Figure prepared by AMEC 2012. Solid red–brown line is extent of resource model.

14.5.2 KPS Stratigraphic Model

A seam model of the KPS stratigraphic unit (which is the most stratigraphically-continuous unit, and is commonly used as the dominant marker horizon) was constructed using the same methodology as the copper resource except that the centroids of KPS composites were used instead of the SMZ composites. However, after the KPS model was constructed, the vertical distances from the centroids of the KPS model blocks to the centroids of the SMZ model were estimated, and the elevations of the KPS model blocks were adjusted using these estimated distances (KPS model was “hung” off of the SMZ model).

14.6 Model Validation

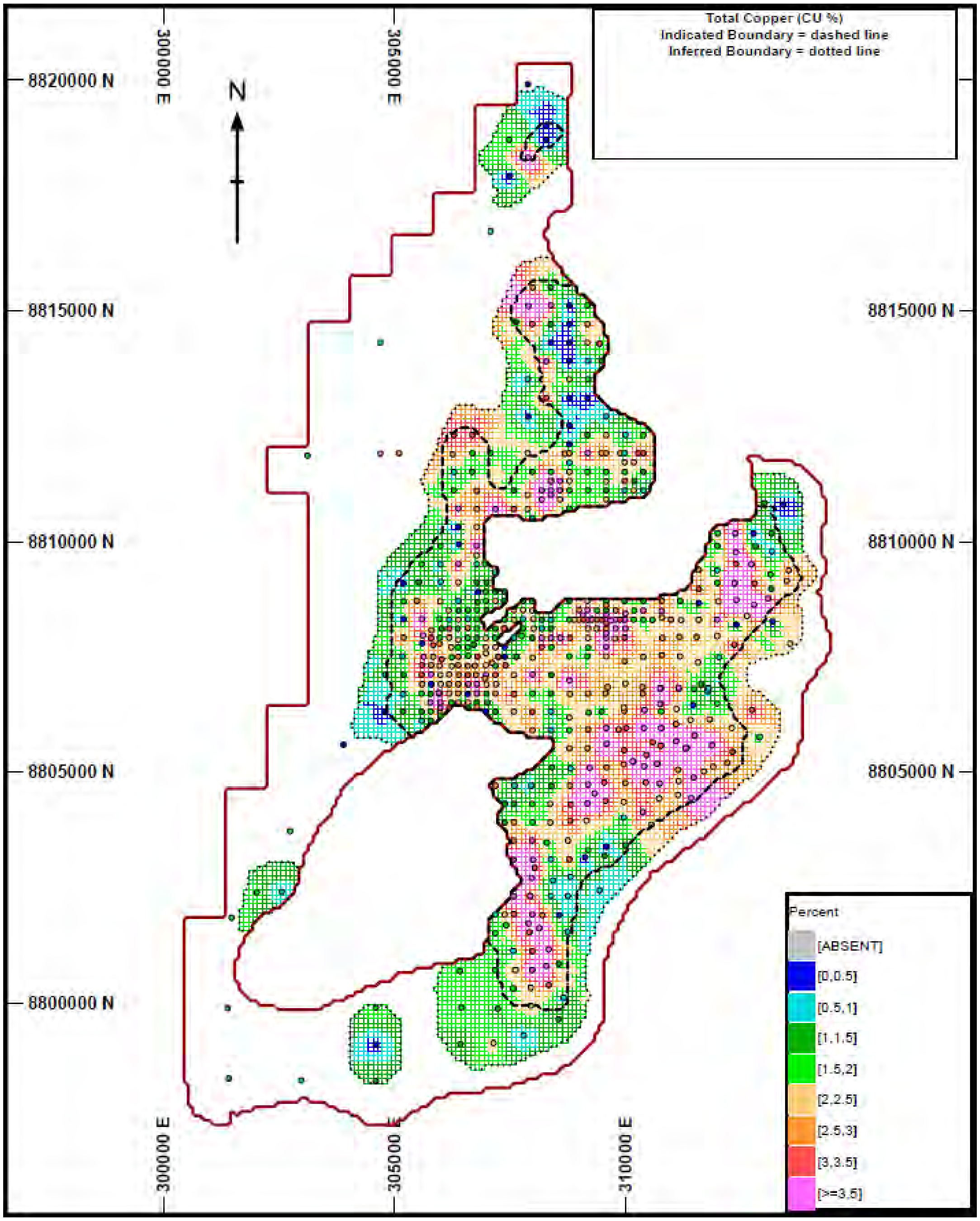
14.6.1 Visual

Estimated block model grades and composite grades were visually examined in plan view and using 3D visualizations. Review of the plans and visualizations showed that the SMZ composites and model blocks agree well. Plan views comparing composites and model values for Cu, ASCu, RATIO, true thickness and Cu multiplied by true thickness are shown in Figure 14-10 through Figure 14-14 respectively.

14.6.2 Nearest Neighbor

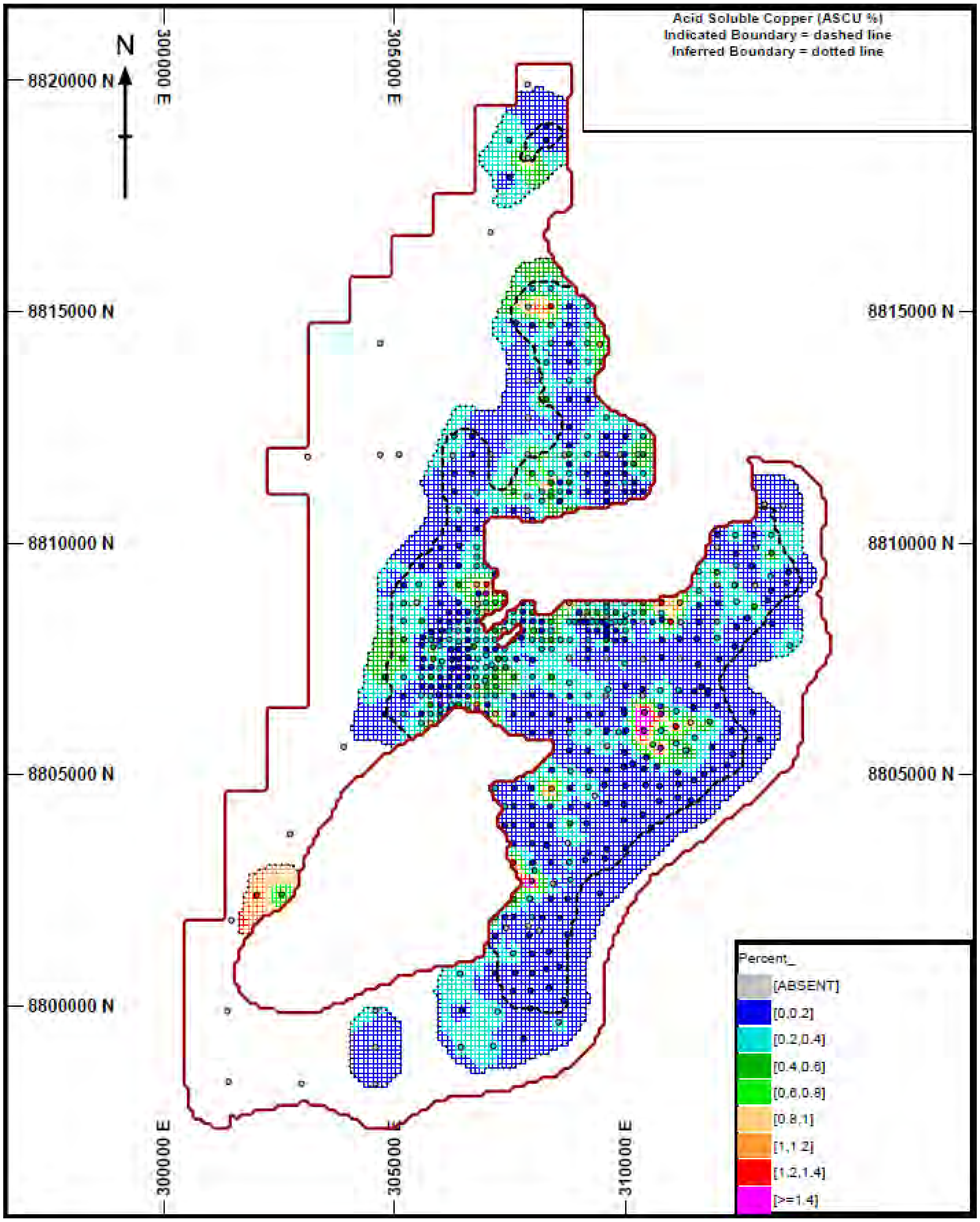
The block model was checked for global bias by comparing the average grade (with no cut-off) from the model with that obtained from nearest-neighbour (NN) model estimates. The nearest-neighbour estimator produces a theoretically globally-unbiased estimate of the average value when no cut-off grade is imposed, and is a good basis for checking the performance of the different estimation methods. The model was validated by comparison of composite and model mean values for copper grade times true thickness, and vertical thickness as indicated in Table 14-4.

Figure 14-10: Total Copper (Cu%, Domain 1)



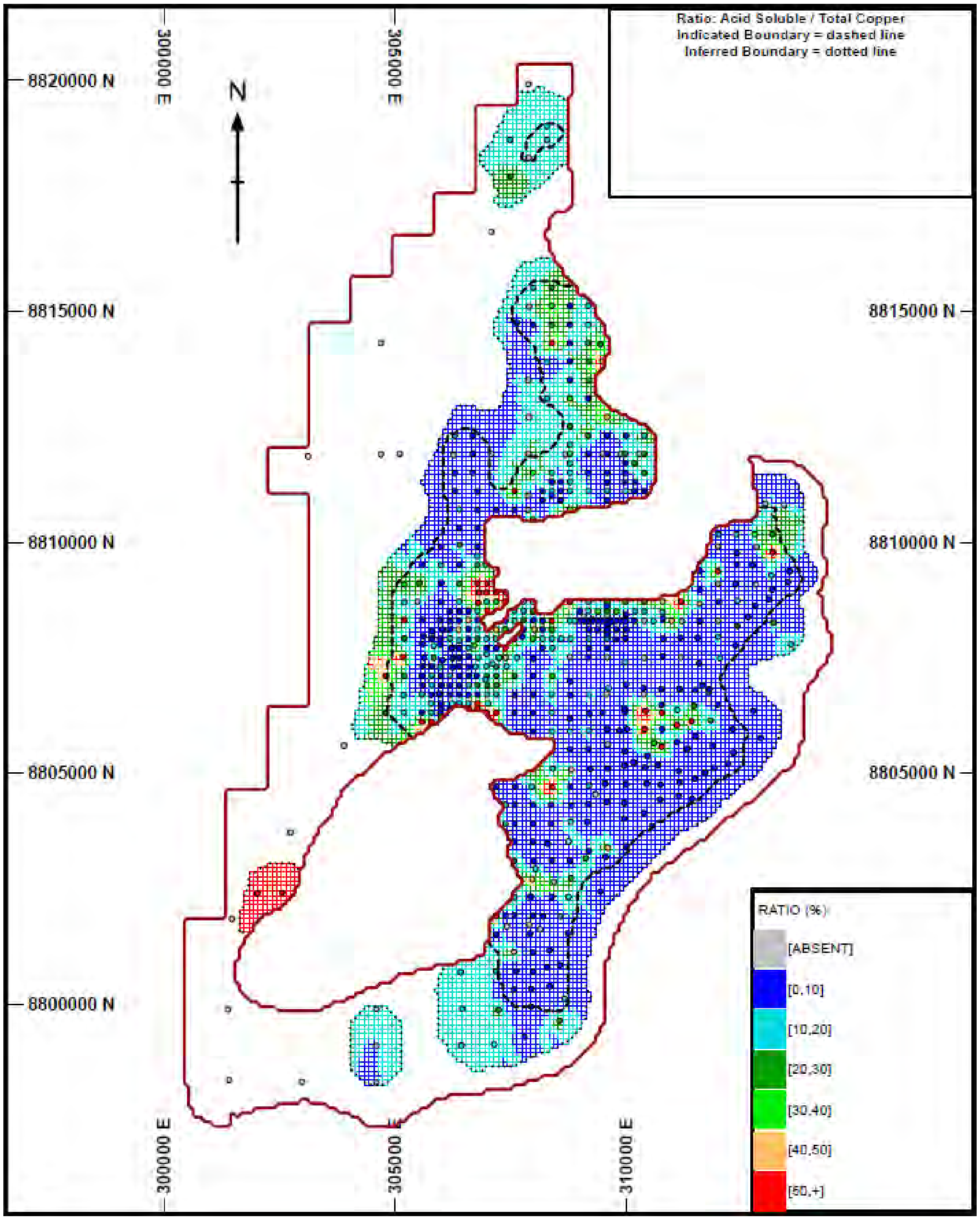
Note: Figure prepared by AMEC 2012. Solid red–brown line is extent of resource model.

Figure 14-11: Acid Soluble Copper (ASCu%, Domain 1)



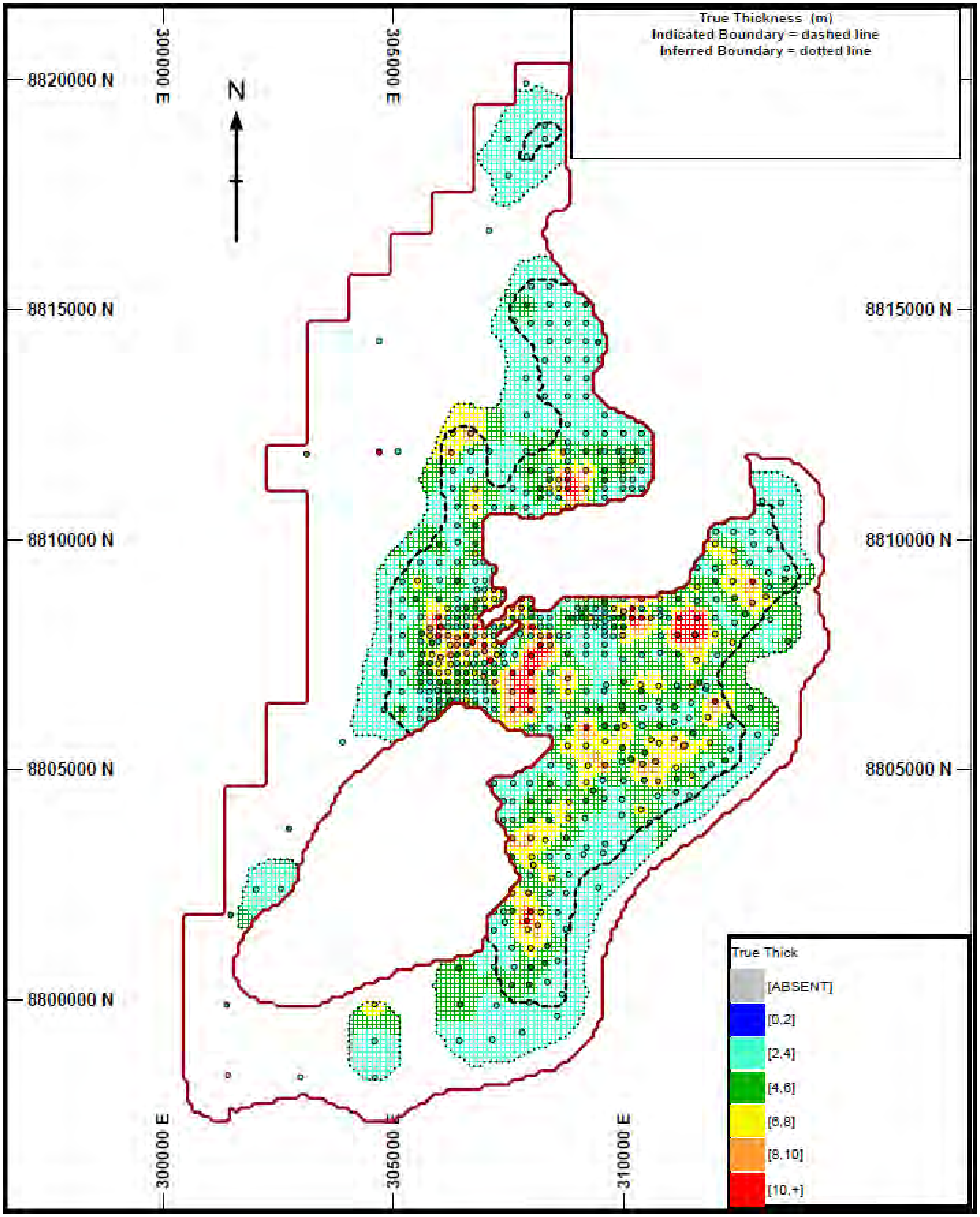
Note: Figure prepared by AMEC 2012. Solid red–brown line is extent of resource model. Note colour scale is different from the Cu scale used in Figure 14-10.

Figure 14-12: Ratio of ASCu/Cu (%)



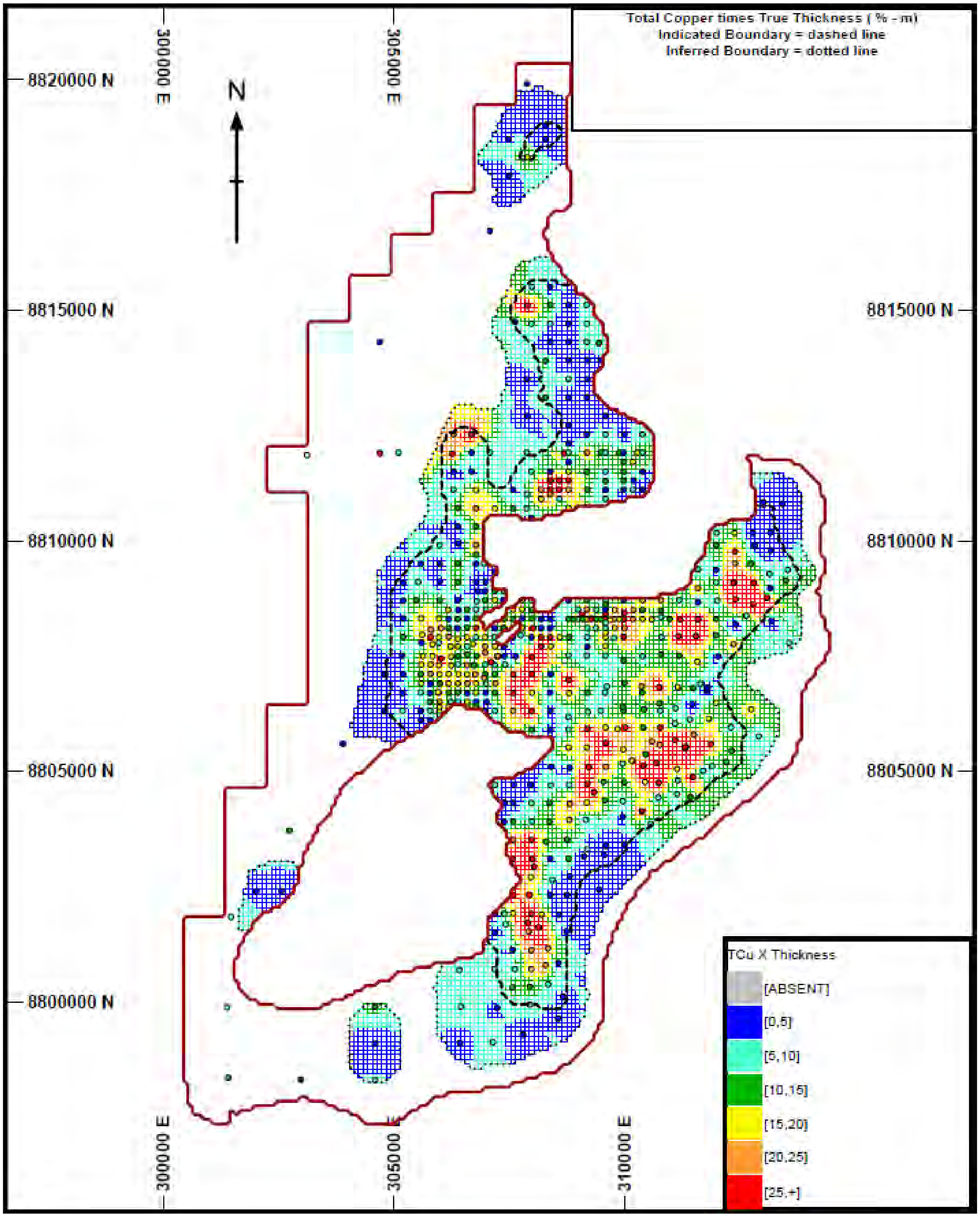
Note: Figure prepared by AMEC 2012. Solid red–brown line is extent of resource model. The ratio does not appear to be related to depth. Locally the ratio exceeds 50%.

Figure 14-13: True Thickness (m, Domain 1)



Note: Figure prepared by AMEC 2012. Solid red–brown line is extent of resource model.

Figure 14-14:Total Copper Times True Thickness (% m, Domain 1)



Note: Figure prepared by AMEC 2012. Solid red–brown line is extent of resource model.

Table 14-4: Mean Grades for Composites and Models (no cut-off applied)

Indicated				
	Composite	**Model ID2**	**Model NN**	**Relative Diff**
CuxTrueThkT	13.94	12.99	12.75	2%
ASCuxTrueThk	1.49 [1]	1.30	1.26	3%
TrueThk	5.34	5.07	5.04	1%
Inferred				
	Composite	**Model ID2**	**Model NN**	**Relative Diff**
CuxTrueThkT	4.80	6.45	5.91	9%
ASCuxTrueThk	0.79 [1]	0.81	0.75	7%
TrueThk	3.47	3.71	3.63	2%

Notes:

1. The composite mean Cu is higher than the nearest-neighbour Cu because data are clustered in high-grade areas
2. Percentage differences are calculated from the original values such that there may be some apparent discrepancies due to rounding

Global biases for the Indicated Mineral Resource domains are generally well below the recommended AMEC guidelines of ±5% (relative difference). Biases greater than the ±5% guidelines do exist for Inferred, and these biases should be studied in preparation for future models.

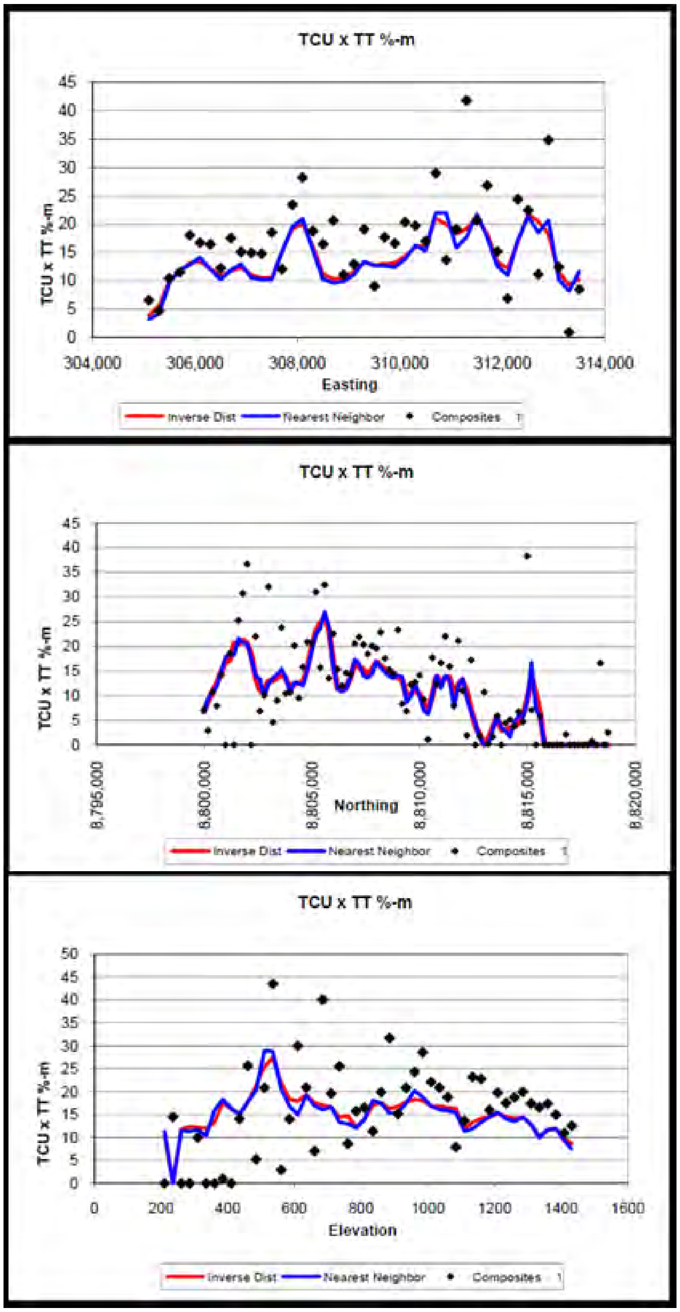
14.6.3 Local Bias Checks (Swath Plots)

Checks for local biases in Domain 1 were performed for Cu time true thickness and true thickness by creating and analyzing local trends in the grade estimates. An example swath plot for Cu times true thickness is shown in Figure 14-15.

These checks are carried out by plotting the mean values from the NN estimate versus the ID2 estimates in east–west, north–south and vertical swaths or increments. Swath intervals were 200 m in both the northerly and easterly directions, and 25 m vertically.

The lines shown on the swath plots represent the average grades; the red line represents the ID2 model grades and the blue line represents the NN model grades; the black dots represent the composite grades. Because the NN model is declustered, it is a better reference model to validate the ID2 resource model. Composites are not declustered, and thus they only provide an indicative check. In general, no local biases were noted.

Figure 14-15: Swath Plots for Total Copper Times True Thickness (% m)



Note: Figure prepared by AMEC, 2012

14.7 Classification of Mineral Resources

The Mineral Resources have been defined taking into account the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves.

14.7.1 Inferred Mineral Resource

An "Inferred Mineral Resource" is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.

Areas outlined by core drilling at approximately 800 m spacing with a maximum extrapolation distance of 600 m, 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. Locally the SMZ can vary as to stratigraphic position within this classification. In addition, there can be marked differences in individual copper grades between drill holes; this has been seen in other Copperbelt deposits, such as Konkola, Zambia, where there is a mosaic of areas several kilometres in extent having near-constant grade with rapid change in grade at their boundaries over a few hundred metres. At Konkola and Mufulira, Zambia, which are considered analogues for Kamoa, a 500 m drill spacing would be appropriate for declaration of Indicated Mineral Resources. Inferred Mineral Resources that are located adjacent to Indicated Mineral Resources are sometimes, in AMEC's experience, declared based on drill hole spacings of 1 km or 2 km in these deposits. Such spacings cannot prudently be used at Kamoa at this time. There is no mining history, nor detailed knowledge of mineralization controls. Kamoa is believed unique within the DRC in that it is a large deposit that is relatively undeformed in contrast to the smaller "écailles" type deposits, which are fragments uplifted and deformed by salt tectonics such as Tenke-Fungurume.

14.7.2 Indicated Mineral Resource

An "Indicated Mineral Resource" is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as *outcrops, trenches, pits, workings, and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.*

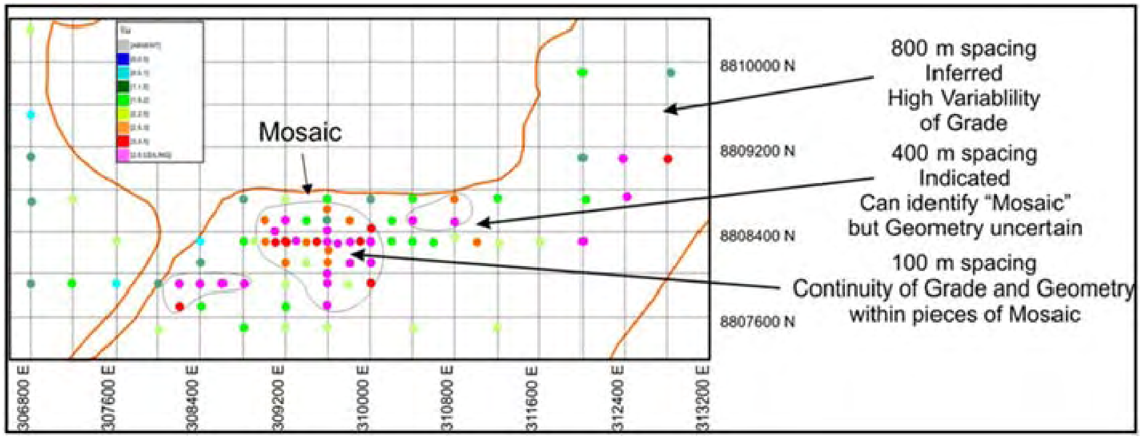
At Kamoa, the 800 m initial drill hole spacing has been locally halved to 400 m. Mineral Resources within the area drilled on a 400 m spacing and which display grade and geological continuity could be considered Indicated. At this spacing, contiguous holes tend to show similar grades and thicknesses (Figure 14-17), but the stratigraphic position of the SMZ can vary (Figure 14-19). The mosaic textures of the distributions of grade and thickness are revealed, but the geometries of mosaic pieces are uncertain. Locally, a closer than 400 m drill hole spacing was used near faults to reflect the uncertainty due to the structural complexity.

14.7.3 Measured Mineral Resource

A "Measured Mineral Resource" is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings, and drill holes that are spaced closely enough to confirm both geological and grade continuity.

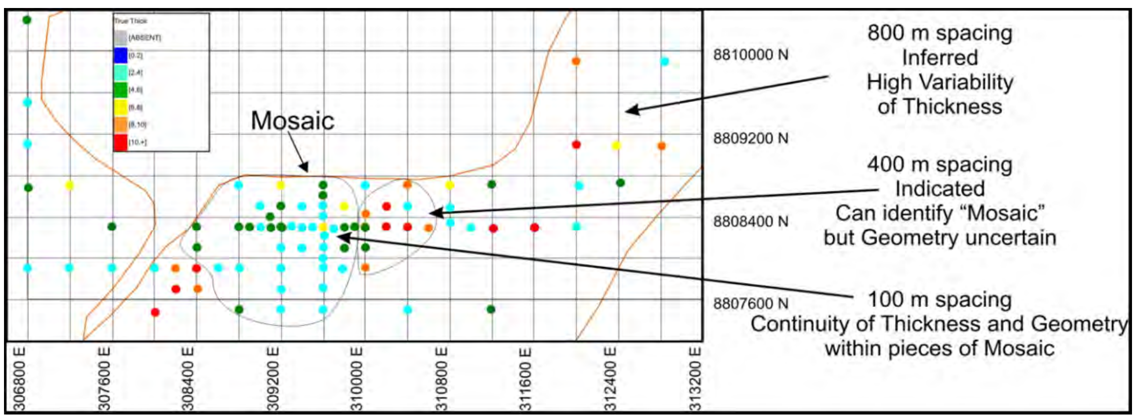
Figure 14-16 and Figure 14-17 show two fences of holes at 100 m spacing. Drill holes on these lines show excellent continuity of grade and thickness, and the stratigraphic position of the SMZ (Figure 14-18) typically varies by only a few metres from core hole to core hole. At a 100 m spacing, it should be possible to define minor faults and their displacements. Therefore, a 100 m spacing is suitable for declaring Measured Mineral Resources.

Figure 14-16: Comments on Mineral Resource Classification Total Copper



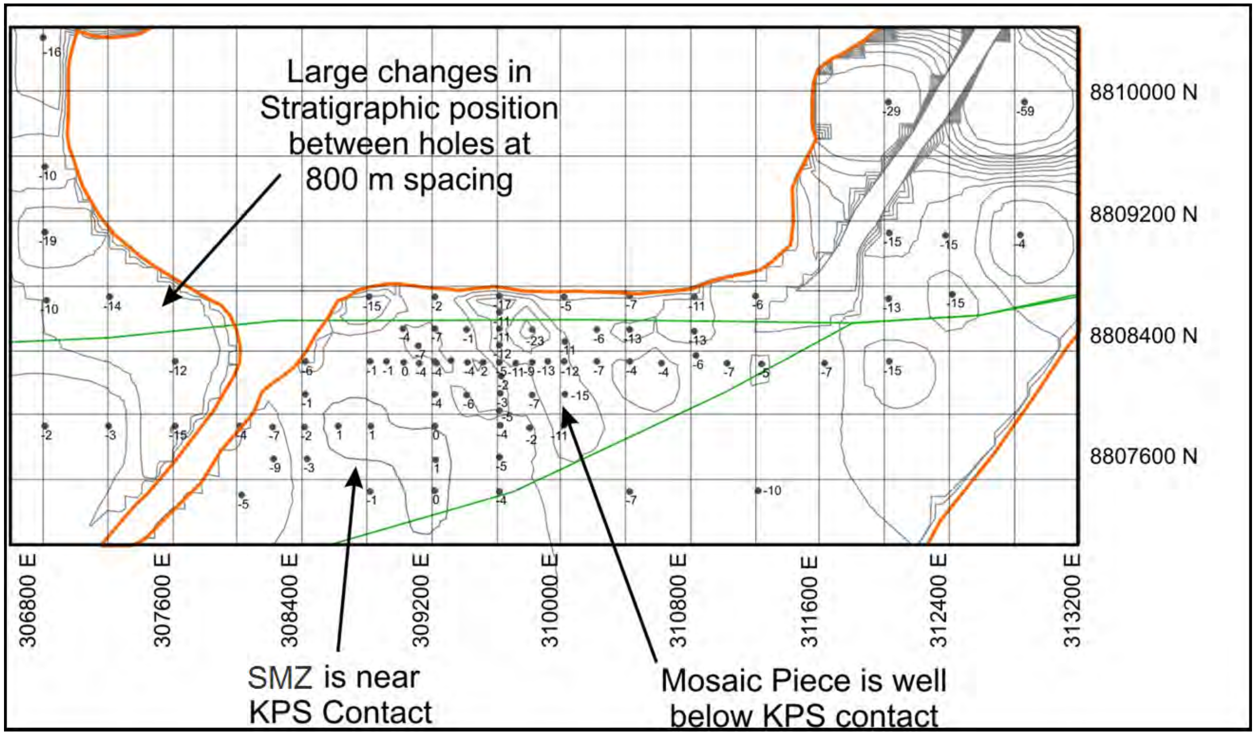
Note: Figure generated by AMEC, 2012

Figure 14-17: Comments on Mineral Resource Classification True Thickness



Note: Figure generated by AMEC, 2012

Figure 14-18: Comments on Mineral Resource Classification, Stratigraphic Position (off-set of SMZ from base of KPS)



Note: Figure generated by AMEC, 2012

In AMEC's experience on the Zambian Copperbelt, 100 m to 120 m is the typical interval between drill cubbies (stations) in hanging wall or footwall drives, and this drilling supports production planning for stope blocks. Additional underground drilling/crosscut chip sampling is done at intervals as close as 15 m for final layout of stopes within the stope blocks.

Blocks classified as Measured Mineral Resources should be surrounded by holes at the requisite spacing. With the current drill hole dataset, this would occur only at the center of the cross of holes drilled on 100 m spacing (one to four blocks). Due to the small number of blocks, no Measured Resources are declared at this time.

The classification should be considered preliminary until more data are gathered and geostatistical analysis can be performed. In addition, the confidence levels required to support mine planning should be evaluated so that some or all of mineralization classified as Measured Mineral Resources and Indicated Mineral Resources can be converted to Proven and Probable Mineral Reserves during future Pre-feasibility/Feasibility Studies.

14.8 Reasonable Prospects of Economic Extraction

AMEC has used a 1% Cu cut-off grade to declare Mineral Resources. 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 mineralization to that identified at Kamoa.

To test the cut-off grade for the purposes of assessing reasonable prospects of economic extraction, AMEC performed a conceptual analysis based on conditions considered appropriate for the region and information available as of January 2013. A copper price of US$3.30/lb and an acid credit of US$300/t was assumed.

The following additional key parameters were used:

- Recovery for Supergene = 86.01(1-exp(-0.9173TCu))
- Recovery for Hypogene = 86.20(1-exp(1.4100TCu))
- Concentrate grades for Supergene 45.1% Cu and 10.7% S
- Concentrate grades for Hypogene 32.8% Cu and 21.7% S
- Mining costs of US$35/t
- Concentrator and tailings costs of US$10/t treated
- Smelting costs of US$90/t of concentrates

- US$500/t transport and refining costs for blister copper
- 2% royalty on NSR – transport costs.

The recovery equations were provided by project metallurgists in December 2012; typical recoveries are shown in Table 14-5.

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 above a nominal 2% copper cut-off in the 2012 PEA (see Section 16.4). There are additional areas for which reasonable prospects for eventual economic extraction exist and which might be scheduled if the nominal 5 Mtpa production rate used for the 2012 PEA was increased to as much as 20 Mtpa. These additional areas are included using a 1% copper cut-off. There is a small percentage (~5%) of tonnage with copper grades between 1.0% and 1.3% that will not cover their full mining costs. Sensitivity analysis using a copper price of US$2.75/lb and US$250/t for acid credits indicates (~11%) of tonnage with copper grades between 1.0% and 1.5% that 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 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 economic extraction.

AMEC 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 10% (see Section 16.4), but will ultimately depend on the ability of the mining to follow the resource boundaries.

14.9 Mineral Resource Statement

The Mineral Resources were classified in accordance with the 2010 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, are the Qualified Persons for the Mineral Resources. Mineral Resources are stated in terms of total copper (Cu). Mineral Resources are reported at a base case total copper cut-off grade of 1% Cu and a minimum thickness of 3 m.

Table 14-5: Typical Recoveries used to Assess Reasonable Prospects for Economic Extraction

Head Grade %TCu	Supergene	Hypogene
1	51.6	65.2
2	72.3	81.1
3	80.5	84.9
4	83.8	85.9
5	85.1	86.1

Indicated and Inferred Mineral Resources were reported for Domain 1 where all blocks have been estimated for Cu and ASCu, refer to Table 14-6.

Table 14-6 summarises the estimated Mineral Resources by resource classification. In total there are 739 Mt of Indicated Mineral Resources at 2.67% Cu and 227 Mt of Inferred Mineral Resources at 1.96% Cu using a 1% Cu cut-off and minimum 3 m thickness.

Table 14-7 shows the sensitivity of the Kamoa Mineral Resource to cut-off grade. A 1% total copper cut-off grade has been traditionally used to declare Mineral Resources and Mineral Reserves on the Zambian Copperbelt, and in AMEC's opinion a similar cut-off is applicable at Kamoa. All of the cases presented represent applications of cut-off grades to blocks in the resource model that was constructed with mineralized intercepts using a 1 % Cu cut-off grade over a 3 m minimum length. Within an intercept the grade, although meeting a 1 % Cu cut-off, can be quite variable, and changing the SMZ intercepts to reflect higher cut-offs is not advisable.

As an additional study, mineralization above a 2.0% and a 2.5% Cu cut-off was plotted separately to provide an indication of the continuity of the Mineral Resources if a higher copper cut-off grade was to be applied. Figure 14-19 (figure on left) shows that the mineralization greater than 2.0% Cu (which approximates the cut-off used in the PEA) remains relatively contiguous and therefore the deposit at this cut-off grade is amenable to forming underground mineable stopes. The figure on the right in Figure 14-19 shows that if a high-grade cut-off of 2.5% is applied, the continuity of the mineralization remains relatively contiguous.

The sensitivity case using a 2.0% copper cut-off, which equates to the cut-off grade used in the PEA, has Indicated Mineral Resources of 550 Mt at 3.04% Cu and Inferred Mineral Resources of 93 Mt at 2.64% Cu. The high-grade sensitivity case using a 2.5% copper cut-off grade has Indicated Mineral Resources of 377 Mt at 3.40% Cu and Inferred Mineral Resources of 51 Mt at 2.97% Cu.

Table 14-6: Kamoa Project Indicated and Inferred Mineral Resource for Domain 1 (at 1% Cu cut-off grade)

Category	Tonnage (Mt)	Area (km^2)	Cu (%)	True Thickness (m)	Contained Copper (kt)	Contained Copper (billion lbs)
Indicated	739	50.5	2.67	5.20	19,700	43.5
Inferred	227	20.5	1.96	3.84	4,460	9.8

Notes:

1. Mineral Resources have an effective date of December 10, 2012. Harry M. Parker and Gordon Seibel, both SME Registered Members, are the Qualified Persons responsible for the Mineral Resource estimates. The Mineral Resource estimate was prepared by Mr. Seibel.
2. Mineral Resources are reported using a total copper (Cu) cut-off grade of 1% Cu and a minimum assumed thickness of 3 metres. A 1% Cu cut-off grade is typical of analogue deposits in Zambia. There are reasonable prospects for economic extraction under assumptions of a copper price of US$3.30/lb; sulphuric acid credits of US$300/t of acid produced; employment of underground mechanized room-and-pillar mining methods; and that copper concentrates will be produced and smelted.
3. Reported Mineral Resources contain no allowances for hanging wall or footwall contact boundary loss and dilution. No mining recovery has been applied.
4. The Mineral Resources include the mineralization above a 1% total copper cut-off that is potentially amenable to open pit mining.
5. Tonnage and grade measurements are in metric units. Contained copper tonnes are reported using metric units; contained copper pounds use imperial units.
6. True thickness ranges from 2.4 metres to 17.6 metres for Indicated Mineral Resources and 2.8 metres to 8.4 metres for Inferred Mineral Resources.
7. 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.
8. Tonnages are rounded to the nearest million tonnes; grades are rounded to two decimal places.
9. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.
10. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

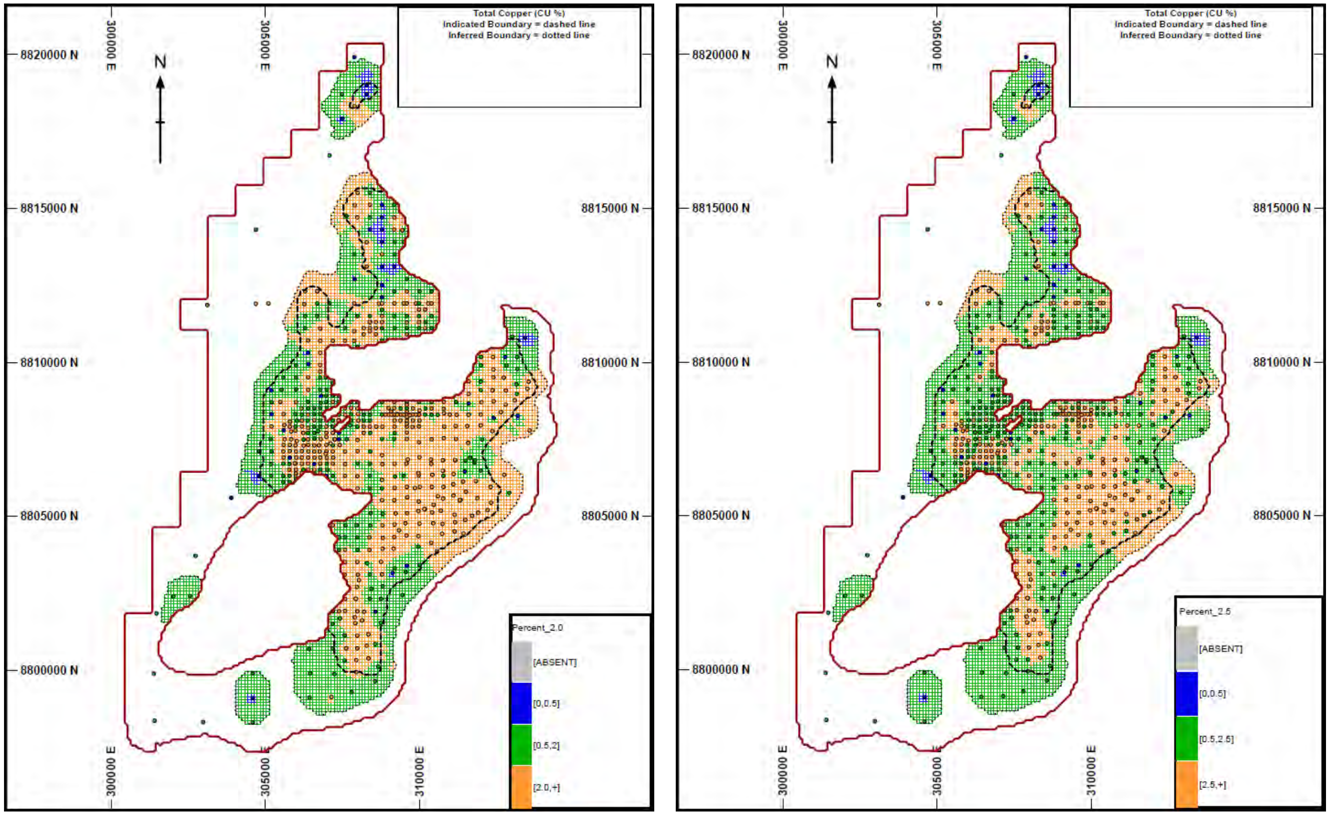
Table 14-7: Sensitivity of Mineral Resources to Cut-Off Grade (base case is highlighted)

Cut-Off %Cu	Tonnage Mt	Area (km^2)	Cu (%)	Contained Copper (kt)	Contained Copper (billion lbs)
Indicated Mineral Resources (cumulative)					
3.00	224	13.6	3.85	8,630	19.0
2.50	377	23.9	3.40	12,800	28.3
2.00	550	34.9	3.04	16,700	36.9
1.75	622	39.9	2.91	18,100	39.8
1.50	675	44.0	2.81	18,900	41.7
1.25	709	47.3	2.74	19,400	42.8
1.00	739	50.5	2.67	19,700	43.5
0.80	755	52.3	2.63	19,900	43.8
0.60	763	53.1	2.61	19,900	44.0
Inferred Mineral Resources (cumulative)					
Cut-Off %Cu	**Tonnage Mt**	**Area (km^2)**	**Cu (%)**	**Contained Copper (kt)**	**Contained Copper (billion lbs)**
3.00	19	1.4	3.40	635	1.4
2.50	51	3.8	2.97	1,520	3.4
2.00	93	7.4	2.64	2,450	5.4
1.75	115	9.5	2.49	2,870	6.3
1.50	164	14.0	2.23	3,670	8.1
1.25	196	17.2	2.10	4,100	9.1
1.00	227	20.5	1.96	4,460	9.8
0.80	249	23.0	1.87	4,660	10.3
0.60	261	24.3	1.82	4,740	10.4

Notes:

1. Base Case 1% copper cut-off is highlighted. Mineral Resources have an effective date of December 10, 2012. Harry M. Parker and Gordon Seibel, both SME Registered Members, are the Qualified Persons responsible for the Mineral Resource estimates. The Mineral Resource estimate was prepared by Mr. Seibel.
2. Mineral Resources are reported using a total copper (Cu) cut-off grade of 1% Cu and a minimum assumed thickness of 3 metres. A 1% Cu cut-off grade is typical of analogue deposits in Zambia. There are reasonable prospects for economic extraction under assumptions of a copper price of US$3.30/lb; sulphuric acid credits of US$300/t of acid produced, employment of underground mechanized room-and-pillar mining methods; and that copper concentrates will be produced and smelted.
3. Tonnages are rounded to the nearest million tonnes; grades are rounded to two decimal places.
4. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.
5. Tonnage and grade measurements are in metric units. Contained copper tonnes are reported using metric units; contained copper pounds use imperial units.

Figure 14-19: Geometry of Higher-Grade Mineralization Defined using a 2.0% Cu Cut-Off (left) and 2.5% Cu Cut-Off (right)



Figures generated by AMEC, 2012. Solid red-brown outline is the resource model extent. Note the legends for each figure are different.

14.10 Factors That May Affect the Mineral Resource Estimate

Factors which may affect the Mineral Resource estimate include:

- Commodity prices and exchange rates.
- Cut-off grade
- Faulting
  - The presence of local faulting could disrupt the productivity of a highly-mechanized operation. In addition, the amount of contact dilution related to local undulations or faulting in the SMZ has yet to be determined. Ivanplats plans to mitigate these risks with further in-fill drilling and an exploratory decline
- Metallurgical recoveries
  - Metallurgical testwork indicates the need for multiple grinding and flotation steps, and variability testwork is yet to be undertaken
- Mining plan
  - Mining recovery could be lower and dilution increased where the dip locally increases on the flanks of the domes. An exploration decline is being considered 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. Location of an exploration decline(s) and the extent of the decline are dependent on further studies
- Infrastructure
  - Exploitation will require building a greenfields project with attendant infrastructure
- Political setting.
  - The DRC is emerging from a period of political instability, and the fiscal and political regime under which mining operations might occur are uncertain. There is provision within the DRC Mining Code for the Government to change the DRC Mining Code and mining rights by decree.

14.11 Exploration Targets

Canadian disclosure standards under NI 43-101 allow the estimated quantities of an Exploration Target to be disclosed as a range of tonnes and grade.

14.11.1 Exploration Targets Adjacent to Indicated and Inferred Mineral Resources

The area inside the model perimeter surrounding the Indicated and Inferred Mineral Resources is shown in Figure 14-20. The ranges of Exploration Target tonnages and grades are summarised in Table 14-8. Tonnages and grades were estimated using an inverse distance to the fifth power for Domain 2 and applying a ±20% variance to the tonnages and grades. In aggregate, these targets could contain from 520 Mt to 790 Mt grading 1.6% Cu to 2.5% Cu.

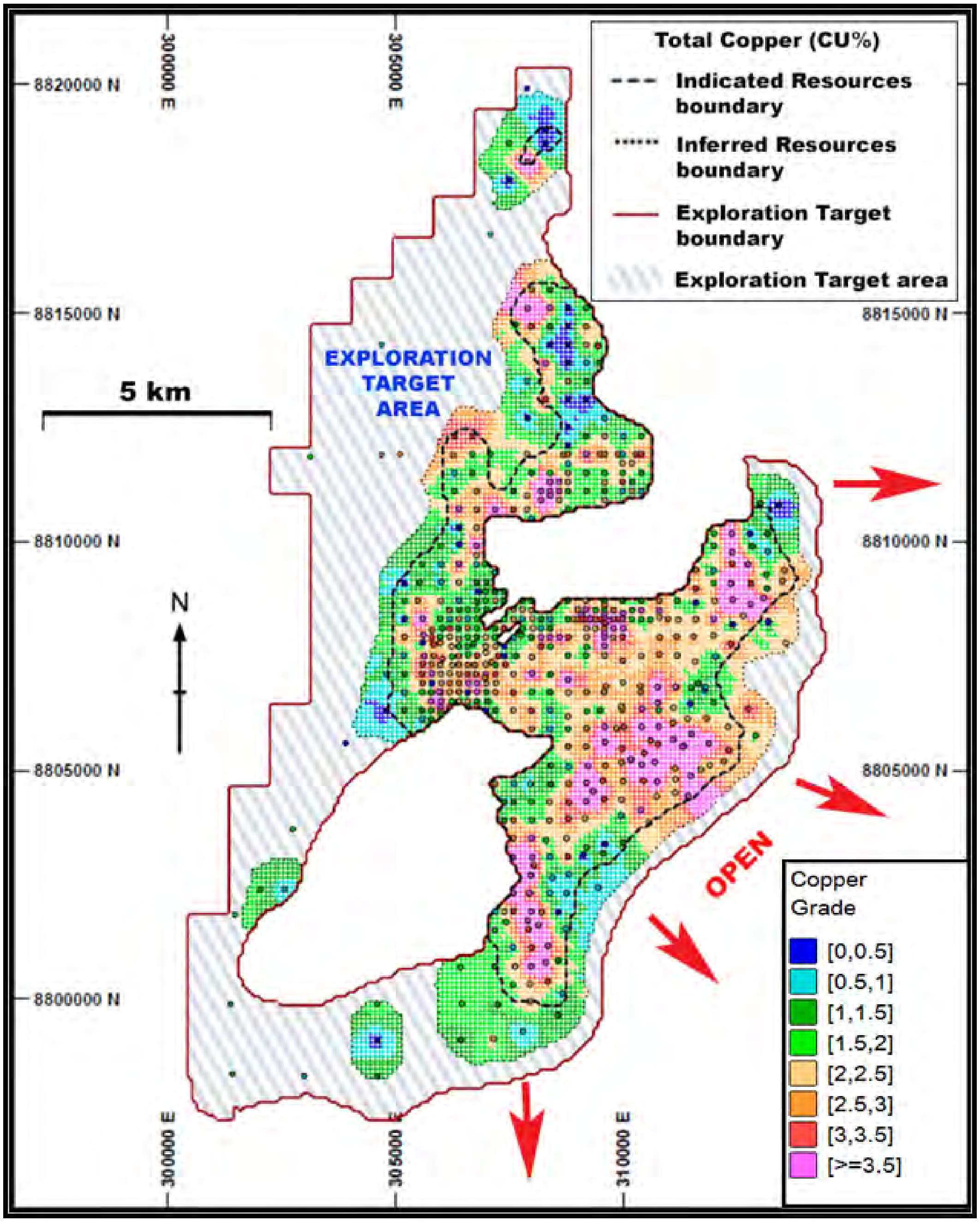
AMEC cautions that the potential quantity and grade of Exploration Targets are conceptual in nature, and that it is uncertain if additional drilling will result in the Exploration Targets being delineated as a Mineral Resource.

14.12 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 drill holes the mineralization and the deposit are untested and open to expansion, even beyond the Exploration Target defined in Section 14.11.

Other exploration prospects exist along strike to the south where additional copper-in-soil anomalies are associated with footwall domes (Kakula and Kakula Northeast) analogous to the Kamoa and Makalu domes (Figure 14-21). There is insufficient information to project a range of tonnage and grade for these exploration prospects although some of the area that has been drill tested (four drill holes) have identified thick, low-grade mineralization with similar stratigraphy to that around the Makalu dome. There is still exploration potential over a large area, but it will require wide-spaced drilling to properly test these prospect areas.

Figure 14-20: Location of Exploration Targets and Drill Holes

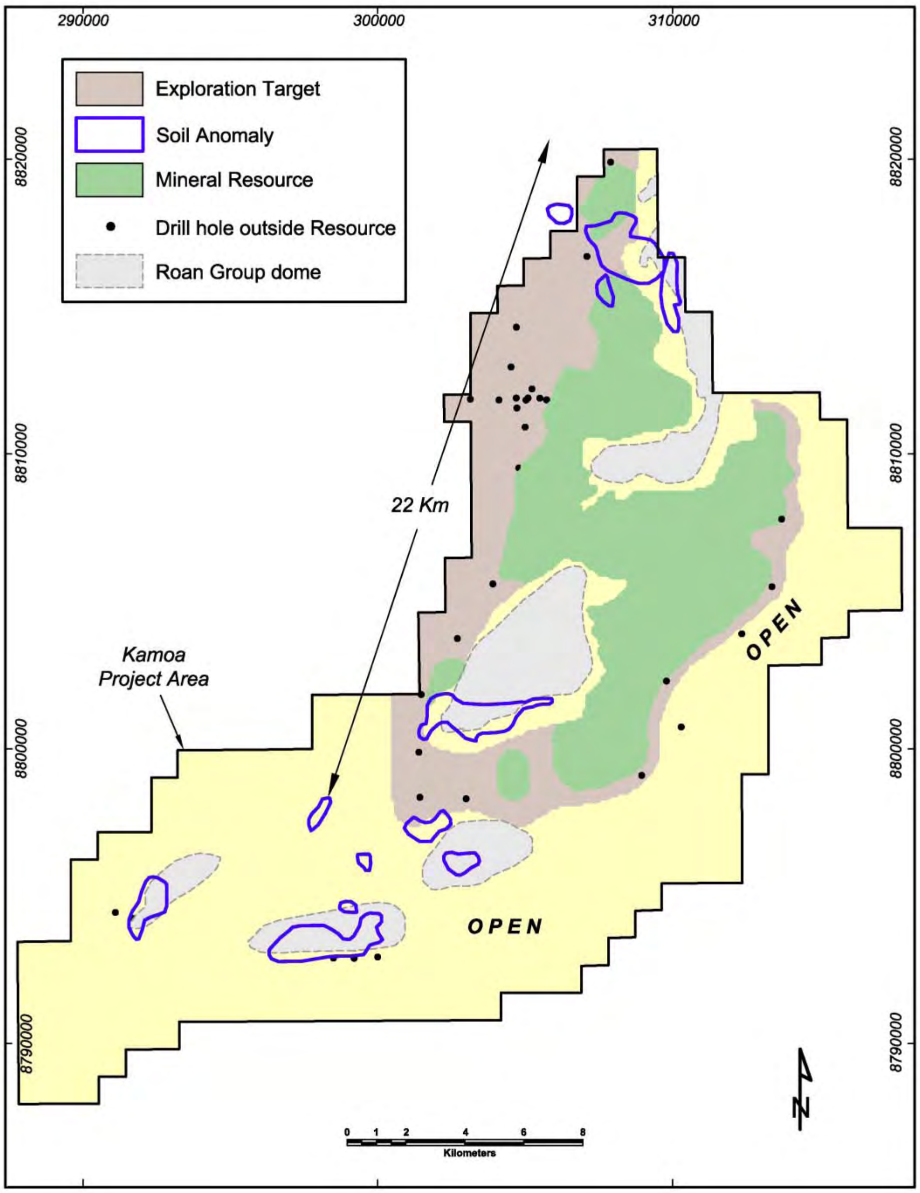


Note: Figure prepared by AMEC, 2013. Solid red–brown line is extent of resource model.

Table 14-8: Tonnage and Grade Ranges for Exploration Targets

Target	Low-range Tonnage Mt	High-range Tonnage Mt	Low-range Grade (%Cu)	High-range Grade (%Cu)
Total	520	790	1.6	2.5

Figure 14-21: Map showing Areas of Additional Exploration Potential



Note: Figure courtesy Ivanplats, modified by AMEC, 2013.

14.13 Other Models and/or Tabulations

14.13.1 Open Pit Seam Models

Two seam models were constructed below and above over the entire extent of the underground model using a 0.5% Cu cut-off to help evaluate the open pit potential for future studies. The 0.5% Cu skins were constructed by Gordon Seibel of AMEC as follows:

- The lengths of the SMZ composite used for the underground model were extended both above and below by lowering the cut-off from 1% to 0.5 %. The extended composited were then flagged as SMZ_OP=1 above the underground model (hanging wall), SMZ_OP=2 (identical to the underground model) or, SMZ_OP=3 below the underground model (footwall).
- Vertical thickness of composites above and below the underground model were calculated by dividing the true thickness of the composite by the cosine of the dip of the SMZ. If no assays greater than 0.5% were adjacent to the SMZ, a composite was created with a vertical thickness of zero and null grades.
- Vertical thickness, total copper, and sulphur were estimated into the underground model for both the hanging wall and foot wall composites using a length-weighed inverse distance to the second power. Acid soluble assays were then calculated by multiplying the estimated total copper by the acid soluble:total copper (ASCu:TCu) ratio estimated into the underground model.
- The vertical thicknesses in the underground model were then used to construct a hanging wall skin above, and a foot wall skin below the underground model with block heights set to the estimated vertical thickness.
- The estimated total copper, acid soluble copper and sulphur for the seam model above and the seam model below were assigned to the corresponding newly constructed seam models.
- The 100 x 100 m underground model and 0.5% skins were then split into sixteen 25 x 25 m blocks and projected to the SMZ surface to increase the vertical resolution of the model for open-pit studies.

Table 14-9 shows the additional Mineral Resources if the 0.5% Cu skins adjacent to the underground resource model are mined using an open pit mining method. The 100 m depth was selected as the reporting depth for this mineralization. In AMEC's experience, open-pit mining operations in the DRC and Zambia typically reach this depth.

Table 14-9: Additional Resource Adjacent to Underground Model if Mined in an Open Pit using a 0.5% Cut-Off

Depth Below Topo (m)	Tonnage Mt	Cu (%)	Contained Copper (kt)	Contained Copper (billion lbs)
Indicated Mineral Resources (cumulative)				
25	1.1	0.71	8	0.0
50	28	0.70	197	0.4
75	60	0.70	418	0.9
100	96	0.69	660	1.5
125	121	0.69	836	1.8
150	146	0.69	1,010	2.2
175	173	0.69	1,200	2.6
200	197	0.69	1,360	3.0
Inferred Mineral Resources (cumulative)				
Depth Below Topo (m)	**Tonnage Mt**	**Cu (%)**	**Contained Copper (kt)**	**Contained Copper (billion lbs)**
25	0.4	0.58	2	0.0
50	0.5	0.60	3	0.0
75	2	0.67	16	0.0
100	10	0.69	66	0.1
125	19	0.68	130	0.3
150	36	0.66	234	0.5
175	51	0.66	333	0.7
200	69	0.66	453	1.0

Notes:

1. Mineral Resources have an effective date of January 12, 2013. Harry M. Parker and Gordon Seibel, both SME Registered Members, are the Qualified Persons responsible for the Mineral Resource estimates. The Mineral Resource estimate was prepared by Mr. Seibel.
2. Base Case of a maximum pit depth of 100 m highlighted. There are reasonable prospects for economic extraction because analogue open pits in Zambia and the DRC with similar grades and economics exist with a similar depth.
3. Mineral Resources are reported using a total copper (Cu) cut-off grade of 0.5% Cu and no minimum thickness. A 0.5% Cu cut-off grade is typical of open pit deposits in the DRC.
4. Tonnages are rounded to the nearest million tonnes; grades are rounded to two decimal places.
5. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.
6. Tonnage and grade measurements are in metric units. Contained copper tonnes are reported using metric units; contained copper pounds use imperial units.

Figure 14-22 shows the sensitivity of the underground model tonnes and grade to depth below the surface, as no underground-open pit trade-off studies are available at this time.

14.13.2 Supergene/Hypogene

Supergene and hypogene metallurgical types were tabulated using perimeters derived from the logged mineral codes and shown in Figure 14-23 and tabulated at different cut-off grades in Table 14-10 (supergene) and Table 14-11 (hypogene). These tables are re-tabulations of the mineralization previously reported, and are not additive to those tables.

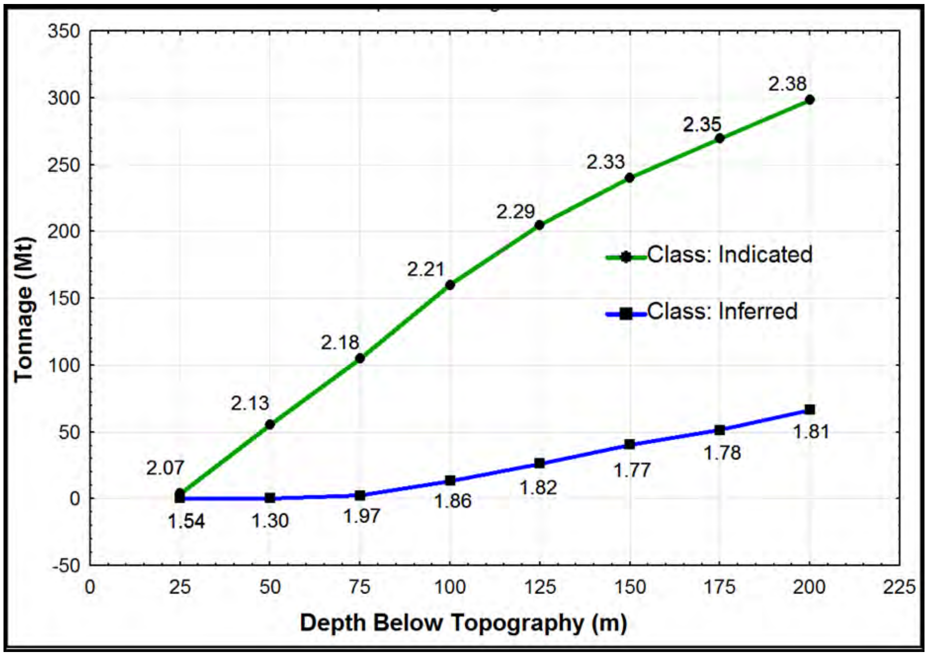
14.14 Comparison of Updated Mineral Resource Estimate to the September 2011 Estimate Supporting the 2012 PEA

The Mineral Resources used for the 2012 PEA were those with an effective date of 24 September 2011 as follows:

- Total Indicated Mineral Resource estimate for the Kamoa deposit of 348 Mt grading 2.64% Cu at a cut-off grade of 1% Cu, over a minimum vertical thickness of 3 m. Based on the application of a 2% Cu cut-off grade over the same minimum thickness criteria, the Indicated Mineral Resource totals 253 Mt grading 3.02% Cu. An approximate cut-off grade of 2% was used for that subset of the Mineral Resources considered in the Preliminary Economic Assessment.
- Total Inferred Mineral Resource estimate for the Kamoa deposit of 462 Mt grading 2.72% Cu at a cut-off grade of 1% Cu, over a minimum vertical thickness of 3 m. Based on the application of a 2% Cu cut-off grade over the same minimum thickness criteria, the Inferred Mineral Resources total 332 Mt grading 3.16% Cu.

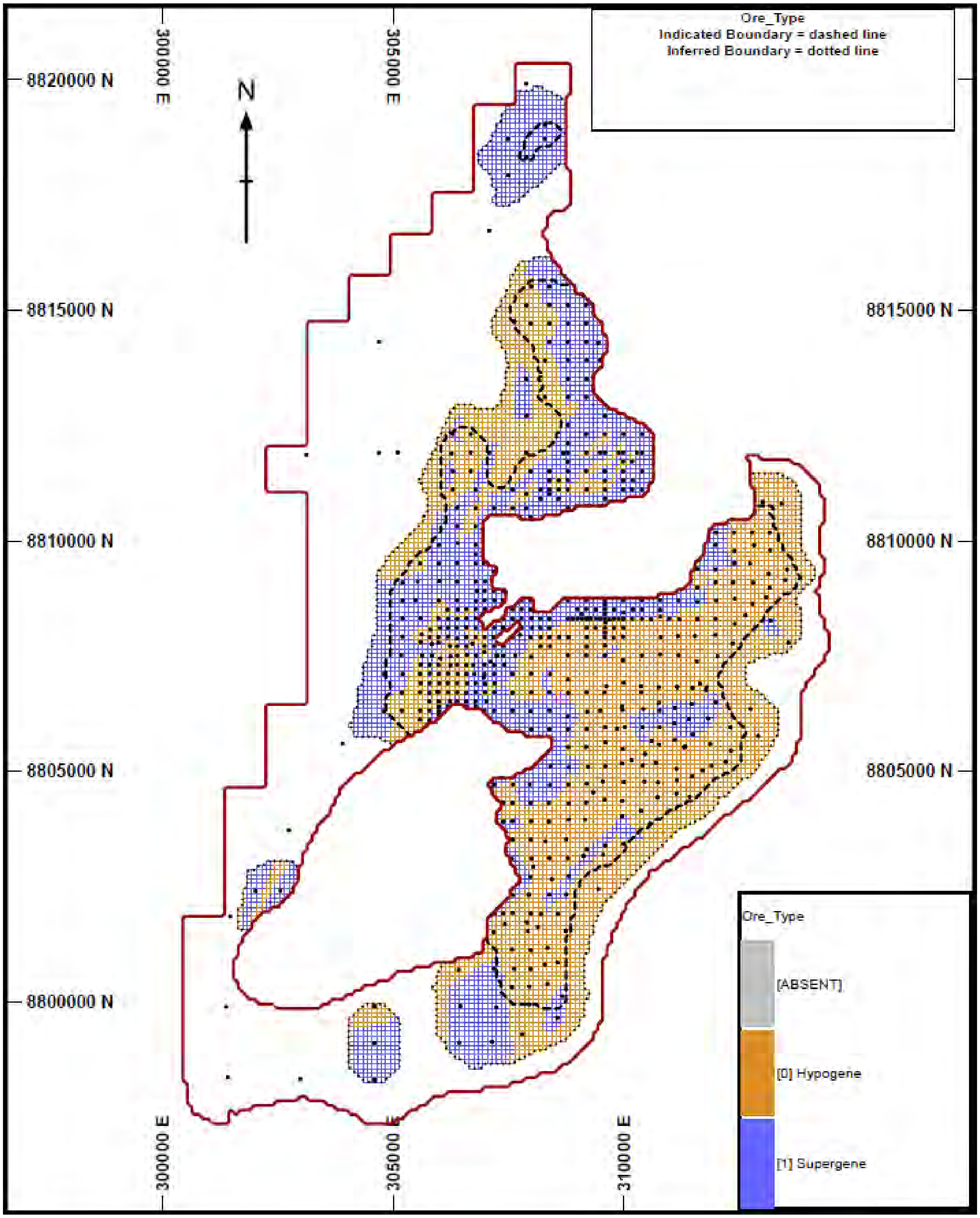
By inspection, the tonnage and grade of the material included in the 2012 PEA mine plan that is based on the September 2011 Mineral Resource estimate is not materially different to the tonnage and grade estimated for the same area in the December 2012 resource update. Figure 14-24 shows the outlines of the Indicated and Inferred classifications for the two estimates.

Figure 14-22: Depth below Topography versus Cumulative Tonnes and Total Copper Grade (for underground model using a 1% Cu cut-off)



Note: Figure prepared by AMEC, 2013.

Figure 14-23: Mineralized Material Classification Types



Note: Figure prepared by AMEC, 2012. Solid black line is extent of the underground resource model.

Table 14-10: Sensitivity Case: Mineral Resource Categorized as Supergene at Different Cut-Off Grades (base case is highlighted)

Supergene: Indicated Mineral Resources (cumulative)					
Cut-Off %Cu	**Tonnage Mt**	**Area (km^2)**	**Cu (%)**	**Contained Copper (kt)**	**Contained Copper (billion lbs)**
3.00	38.6	2.93	3.92	1,510	3.3
2.50	80	5.77	3.31	2,640	5.8
2.00	144	10.1	2.83	4,060	9.0
1.75	182	12.9	2.63	4,770	10.5
1.50	213	15.3	2.48	5,290	11.7
1.25	238	17.7	2.36	5,630	12.4
1.00	255	19.5	2.28	5,820	12.8
0.80	264	20.5	2.24	5,910	13.0
0.60	270	21.1	2.20	5,940	13.1
Supergene: Inferred Mineral Resources (cumulative)					
Cut-Off %Cu	**Tonnage Mt**	**Area (km^2)**	**Cu (%)**	**Contained Copper (kt)**	**Contained Copper (billion lbs)**
3.00	2.4	0.22	3.46	82	0.2
2.50	4	0.43	3.13	137	0.3
2.00	9	0.96	2.64	248	0.5
1.75	16	1.61	2.33	363	0.8
1.50	44	4.33	1.88	826	1.8
1.25	59	5.87	1.76	1,030	2.3
1.00	74	7.5	1.63	1,200	2.6
0.80	83	8.55	1.55	1,280	2.8
0.60	91	9.5	1.47	1,340	3.0

Table 14-11: Sensitivity Case: Mineral Resources Categorized as Hypogene at Different Cut-Off Grades (base case is highlighted)

Hypogene: Indicated Mineral Resources (cumulative)

Cut-Off %Cu	Tonnage Mt	Area (km^2)	Cu (%)	Contained Copper (kt)	Contained Copper (billion lbs)
3.00	185.0	10.7	3.84	7,110	15.7
2.50	298	18.1	3.42	10,200	22.4
2.00	406	24.8	3.11	12,700	27.9
1.75	440	27	3.02	13,300	29.3
1.50	462	28.7	2.96	13,600	30.1
1.25	471	29.6	2.92	13,800	30.4
1.00	484	31	2.88	13,900	30.7
0.80	491	31.8	2.85	14,000	30.8
0.60	493	32	2.84	14,000	30.8

Hypogene: Inferred Mineral Resources (cumulative)

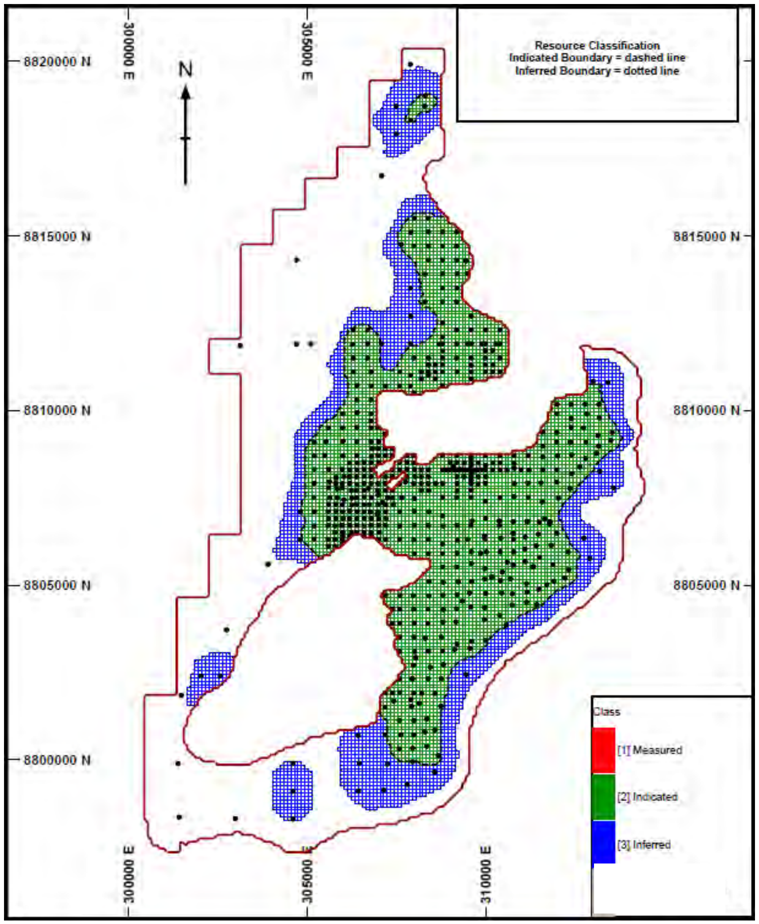
Cut-Off %Cu	Tonnage Mt	Area (km^2)	Cu (%)	Contained Copper (kt)	Contained Copper (billion lbs)
3.00	16.3	1.18	3.40	553	1.2
2.50	47	3.38	2.96	1,380	3.1
2.00	83	6.4	2.64	2,200	4.9
1.75	100	7.87	2.51	2,510	5.5
1.50	120	9.66	2.36	2,840	6.3
1.25	137	11.3	2.24	3,080	6.8
1.00	153	13	2.12	3,260	7.2
0.80	166	14.4	2.03	3,370	7.4
0.60	169	14.8	2.01	3,400	7.5

Notes to accompany Tables 14-10 and 14-11:

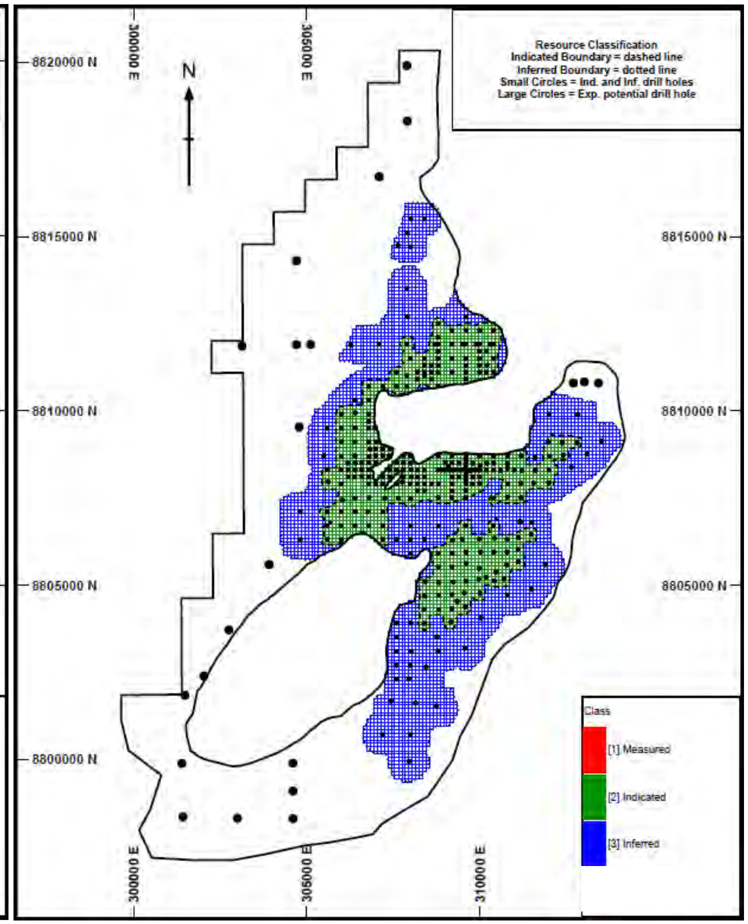
1. Base Case 1% copper cut-off is highlighted. Harry M. Parker and Gordon Seibel, both SME Registered Members, are the Qualified Persons responsible for the Mineral Resource estimates. The Mineral Resource estimate was prepared by Mr. Seibel.
2. Mineral Resources are reported using a total copper (Cu) cut-off grade of 1% Cu and a minimum assumed thickness of 3 metres. A 1% Cu cut-off grade is typical of analogue deposits in Zambia. There are reasonable prospects for economic extraction under assumptions of a copper price of US$3.30/lb; sulphuric acid credits of US$300/t of acid produced, employment of underground mechanized room-and-pillar mining methods; and that copper concentrates will be produced and smelted.
3. Tonnages are rounded to the nearest million tonnes; grades are rounded to two decimal places.
4. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.
5. Tonnage and grade measurements are in metric units. Contained copper tonnes are reported using metric units; contained copper pounds use imperial units.
6. These tables are re-tabulations of the mineralization included in Table 14-7 and are not additive to that table.

Figure 14-24: Comparison of September 2011 and December 2012 Resource Estimate Areas (based on the outlines of Indicated and Inferred Mineral Resources)

December 2012 Resource Estimate



September 2011 Resource Estimate (supports 2012 PEA)



14.15 Comments on Section 14

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 (2010).

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-mechanized operation. In addition, the amount of contact dilution related to local undulations in the SMZ has yet to be determined. Ivanplats plans to mitigate these risks with further in-fill drilling.
- Assumptions used to generate the data for consideration of reasonable prospects of economic extraction. Mining recovery could be lower and dilution increased where the dip locally increases on the flanks of the domes. An exploration decline is being considered 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. Location of an exploration decline(s) and the extent of the decline are dependent on further studies.
- Long-term commodity price assumptions.
- Long-term exchange rate assumptions.
- Operating and capital assumptions. Exploitation will require building a greenfields project with attendant infrastructure.
- Metal recovery assumptions. Metallurgical testwork indicates the need for multiple grinding and flotation steps, and variability testwork is yet to be undertaken.
- The fiscal and political regime under which mining operations might occur are uncertain. There is provision within the 2002 Mining Code for the Government to change the 2002 Mining Code and mining rights by decree and a draft 2012 Mining Code has been circulated. There is also a risk that the DRC Government could change the current royalty, duty, and taxation regime.

15.0 MINERAL RESERVE ESTIMATES

No Mineral Reserves have been estimated for the Project.

16.0 MINING METHODS

16.1 Introduction

This mining section incorporates assumptions, analysis and findings of the 2012 PEA. The mineral resources used in the 2012 PEA mine plan were those with an effective date of 24 September 2011.

The preliminary mine plan presented in this section is partly based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the Preliminary Economic Assessment based on these Mineral Resources will be realized.

The work prepared for Section 16 of the 2012 PEA has not been updated. It is considered that this work remains current as there has not been any additional work or changes to the assumptions that are likely to result in a material change to the conclusions of Section 16 in the 2012 PEA.

The QPs for Section 16 of this Report are Bernard Peters (Section 16.1, 16.3, 16.5), Jarek Jakubec (Section 16.2), and Mel Lawson (Section 16.4).

16.2 Geotechnical

SRK has been involved in the geotechnical and mining aspects at Kamoa since early 2010. This section briefly summarises SRK's previous work on the program.

16.2.1 Kamoa Geotechnical Field Data – Draft (May 2010)

This technical memorandum presents findings from the initial SRK on-site geotechnical review conducted by Jarek Jakubec (April, 2010). The document considers the current geotechnical logging approach, in context to proposed mining activities and recommends changes to the geotechnical data collection system and certain on-site processes.

The document also lays out criteria for the geotechnical grade logging of the KPS (Kamoa pyritic siltstone; Ki1.1.2) unit.

16.2.2 Kamoa Site Visit Report (August, 2010)

This report summarised the Kamoa site visits undertaken by Dr. Wayne Barnett in July, 2010. The objectives of the site visit were to:

1. Review progress in geotechnical characterization and field work recommended by SRK in March 2010.
2. Formulate an opinion on the structural deformation of the deposit and how it could impact the geotechnical characterization of the deposit.

Output from this work included comment on geotechnical logging QA/QC, recommended alterations to data collection procedures and a preliminary 3D structural model.

16.2.3 Geotechnical Evaluation of Kamoa Hanging Wall Rock Conditions (March, 2011)

SRK provided Ivanplats with a geotechnical assessment in PowerPoint® format (December, 2010) of the KPS unit, that was understood to comprise the hangingwall of the Kamoa zone of mineralization. This investigation together with the previous geotechnical data provided information to make recommendations on the extraction ratios for different domains on a scoping level.

Following this work, Ivanplats requested SRK to consider new geotechnical data and conduct a rapid re-evaluation of the previous assumptions and conclusions. SRK was then asked to review the assumption that the KPS forms the hangingwall to the defined resource in all areas. Based on the review of new data, SRK issued updated exclusion percentage values based on the revised hangingwall analysis.

A best-case and worst-case exclusion range was provided to reflect the uncertainty due to the wide-spacing of data and lack of understanding of the structures and associated alteration-deformation zones.

16.2.4 Geotechnical Data Quality Control Site Visits (2011)

SRK completed three site visits to Kamoa during 2011 for the purposes of geotechnical and structural logging QAQC and data quality control.

- June 22–27 – Ross Greenwood, Ryan Campbell, and Desiré Tshibanda completed geotechnical logging QA/QC, training, and data quality review.

- August 5–12 – Ross Greenwood and Desiré Tshibanda completed geotechnical logging QA/QC, reviewed changes implemented to logging practices, and conducted additional data quality reviews.
- August 12–17 – Wayne Barnett 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 memorandums are considered to satisfy tasks within SRK's scoping level proposal issued to Ivanplats on 29 May 2011.

Findings from the visits have been documented in two memorandums which provide outline on-site protocols, quality control reviews, details of the findings, recommendations to future data collection, and update various geotechnical and mining study aspects. Limited on-site data analysis and preliminary findings are also documented.

16.2.5 Structural Data Review

A copy of the Kamoa geotechnical database was provided to SRK in August 2011. SRK had previously checked the detailed structural measurements made on the orientated core. The calculations and database integrity was reviewed with Kamoa personnel until the problems were corrected, and final true structural orientations were then calculated.

The structures were then imported into GoCad software for 3D spatial review. The complete geotechnical logs were also reviewed in the GEMS software, within which the current geotechnical project resides.

A review of fracture frequency data suggests the presence of broad north-east striking brittle strain zones characterized by high fracture frequencies (Figure 16-1). It seems likely that domains of higher and lower fracture frequency will be delineated by the on-going drilling program. It is speculated that the domains are controlled by reactivated basement structures of similar northeast strike, observed as lineaments in the airborne magnetic data images.

The possibility of domains was further tested through the spatial review of the structural orientation measurements. Data from the high fracture frequency domains and low fracture frequency domains was selected and analysed in stereonets to determine the fracture orientation patterns (Figure 16-2).

Figure 16-1: Drill hole Traces displaying Fracture Frequency (Hotter Colors are Higher Frequencies). Possible Corridors of More Intense Fracturing is Highlighted.

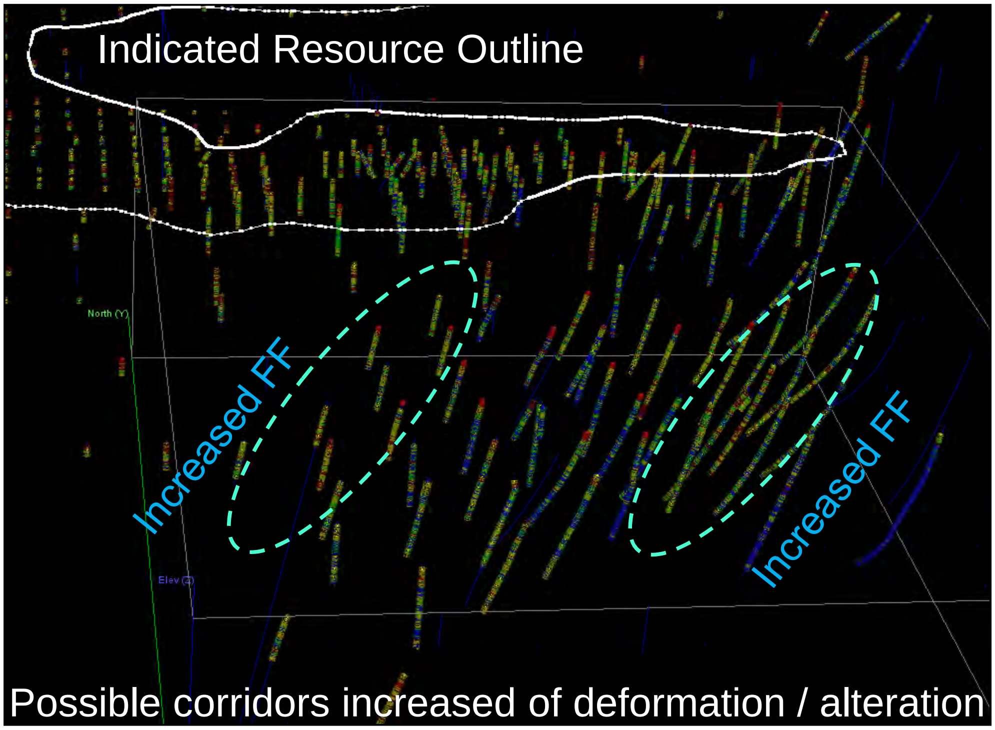
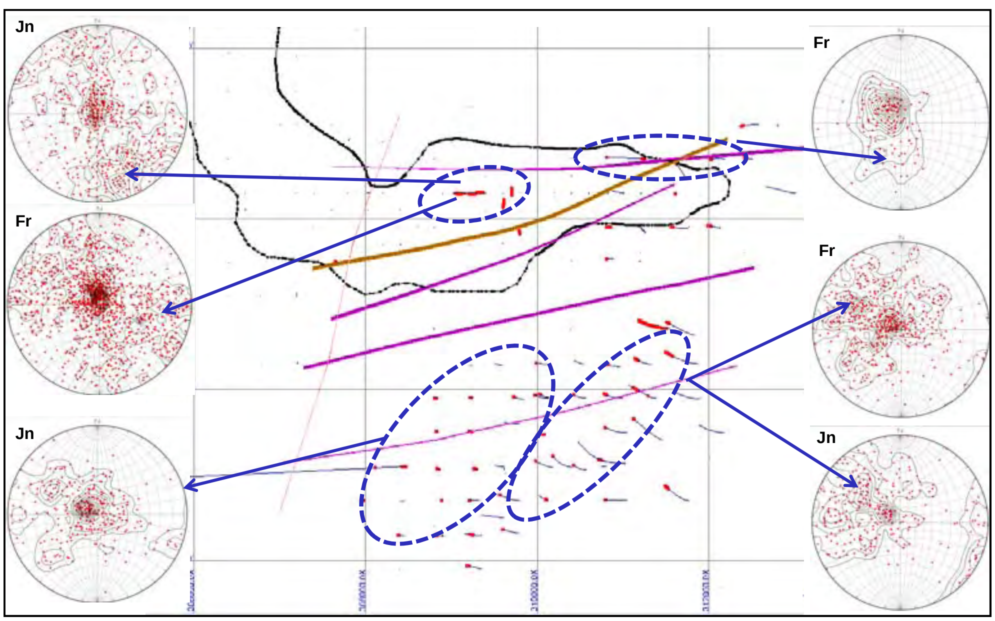


Figure 16-2: Steronet Evaluation of Joint Patterns from Different Localities



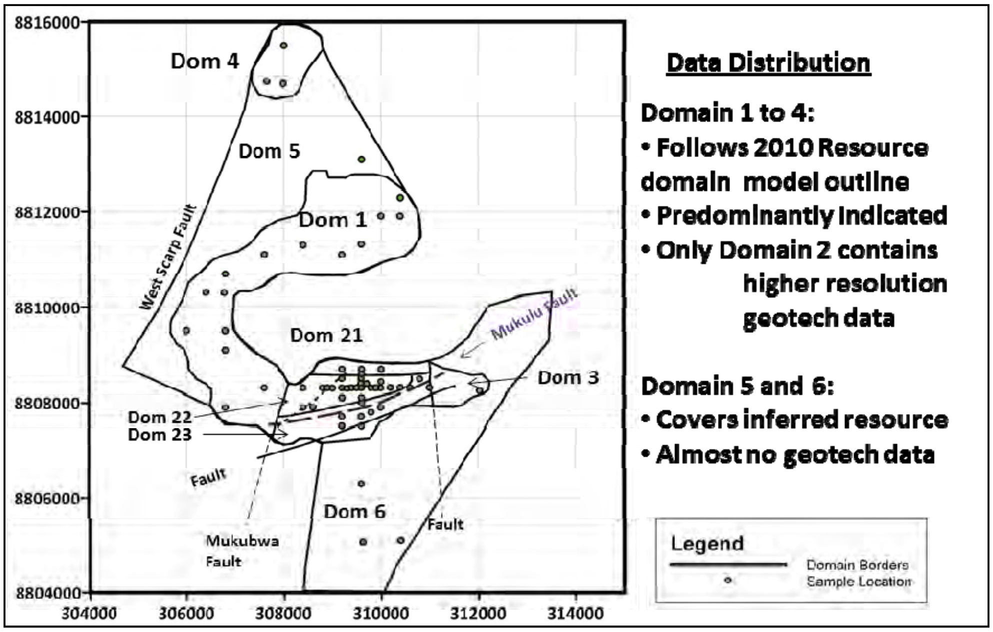
General observations based on the available data are:

- The dominant joint set is the bedding parallel set, which is consistent with core observations.
- The joints in the Kamoa Sud area show more dominant east-west to west-south-west strikes.
- The joints in the Kansoko Central area show dominant northeast strikes. There are also subvertical, south-south-west striking joints strongly represented in the stereonets for both areas.
- All three of these joint set strikes correspond to the interpreted fault patterns across the project area.
- No clear difference in the joint set orientations was observed for the high fracture frequency versus the low fracture frequency domains.

16.2.6 Geotechnical Evaluation

A drill hole database was provided to SRK by African Mining Consultants on 24 February, 2011. Photographs of specific requested drill holes were sent to SRK by African Mining Consultants on the 30 March 2011. Intersection depths with the zone of mineralization were provided to SRK by AMEC on the 29 March 2011. 150 drill holes with KPS specific weathering data were used for the current 2011 analysis utilizing the geotechnical domains shown in Figure 16-3.

Figure 16-3: Geotechnical Domains (SRK, 2010)



16.2.7 KPS Geotechnical Grade

The KPS geotechnical grade was designed to complement geotechnical logging data and visually describe the condition of the KPS unit proximal to the Resource. The following visual criteria were established (May 2010, Figure 16-4):

- KPS Fresh (rating 4) – no weathering or separation of the bedding planes.
- KPS Weathered (rating 3) – no separation of bedding planes but discoloration due to weathering.
- KPS Separated (rating 2) – bedding planes separated on close spacing.
- KPS Disintegrated (rating 1) – core disintegrated with observable strength loss.

Figure 16-4: Representative Core Photos showing KPS Geotechnical Weathering Categories

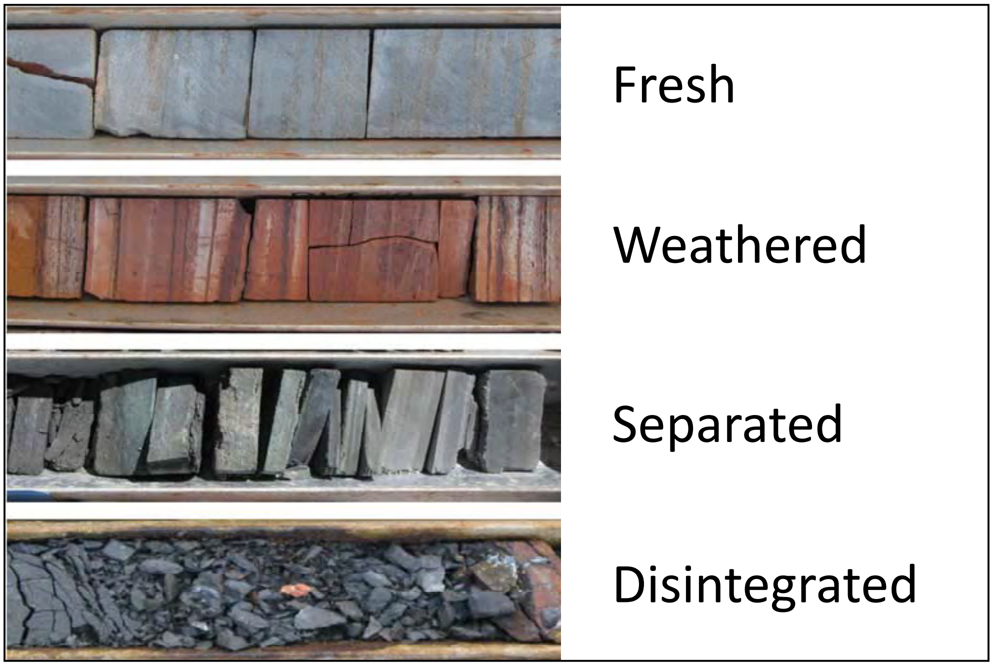
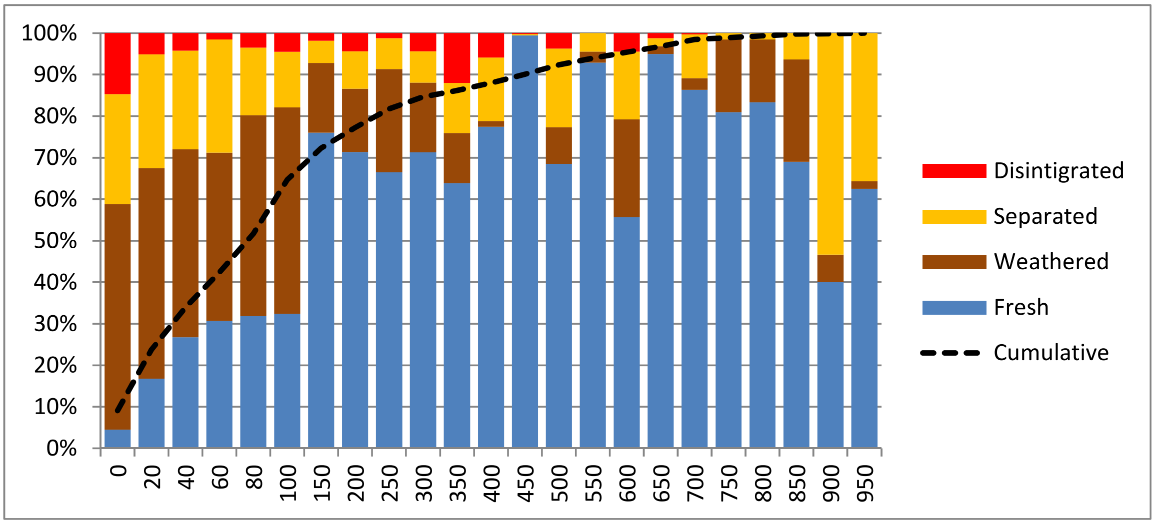


Figure 16-5 (June 2011) illustrates the distribution of the variable KPS geotechnical grades with depth. General trends indicate that the weathering, separated and disintegrated classes decrease with depth while the relative percent of fresh material increases. It should be noted that confidence reduces sharply below 350–400 m depth as illustrated by the cumulative distribution line.

The four hanging wall (HW) quality categories are related to the possible achievable spans, dilution estimates and ground support requirements.

Figure 16-5: Distribution of KPS Geotechnical Grade with Depth (All Holes, ~26,000 0.2 m Composite Records)



16.2.8 Drill hole Assessment

Each drill hole with geotechnical data was evaluated to describe the 10 m of rock located above the mineralization zone. If any part of this 10 m zone consisted of KPS, then specific weathering observation criteria developed by SRK (2010) and implemented by African Mining Consultants were applied to determine the percentage of the 10 m that falls into four weathering categories.

The specific criteria for the weathering categories are as follows:

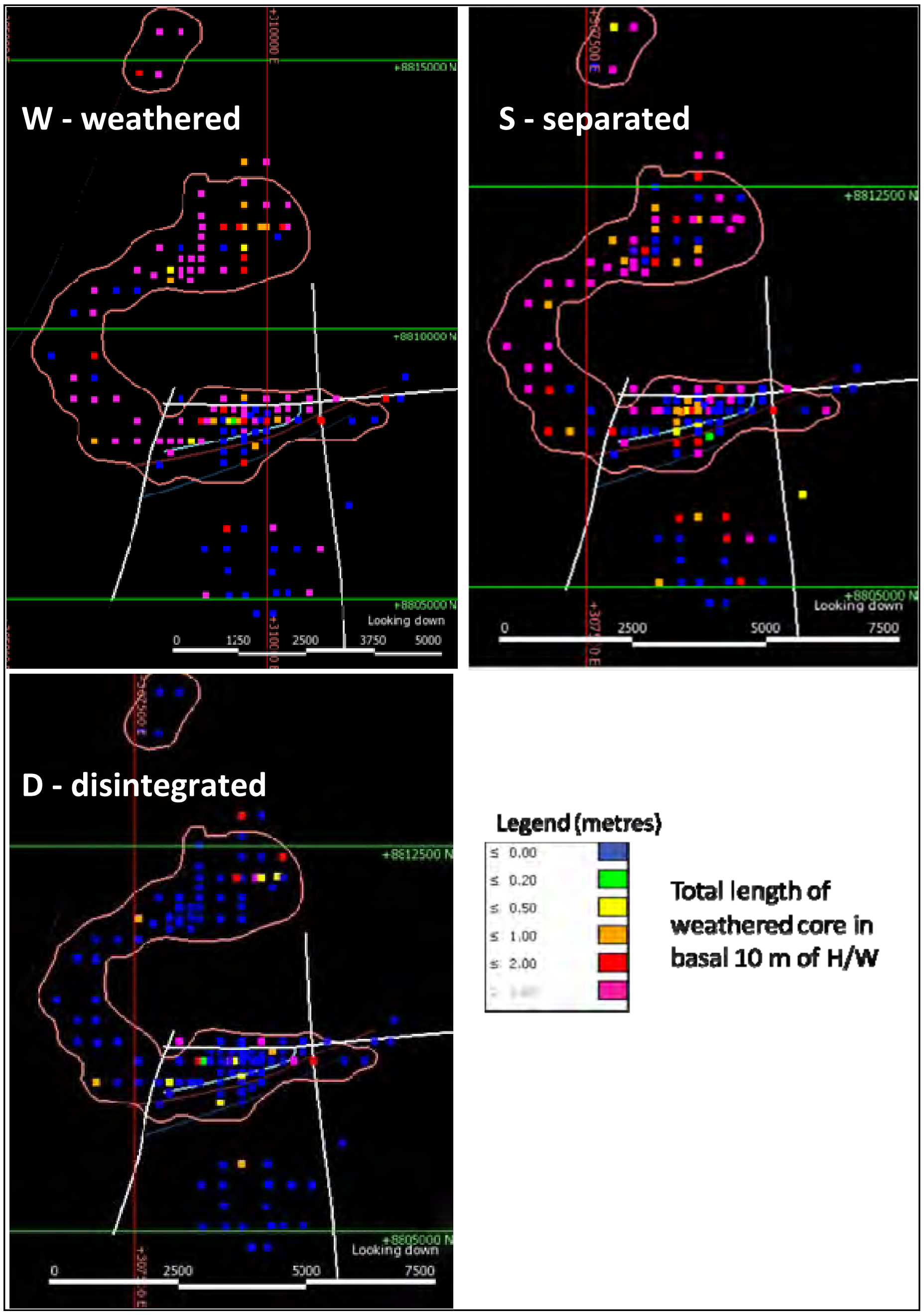
- FR/4 = Fresh HW – no weathering or separation of the bedding planes
- W/3 = Weathered HW – no separation of bedding planes but discoloration due to weathering
- S/2 = Bedding planes separated in HW
- D/1 = Core disintegrated in HW.

If part of the 10 m zone consisted of rock domains not within the KPS, then the geotechnical master table in the drill hole database was cross-queried to get weathering observations. These different weathering observations were related to the above categories (weak weathered = 3, medium and strong weathering = 2, and extremely weathered = 1).

The percentage/length of the 10 m for each category has been imported into Leapfrog (ARANZ Geo Ltd.) for visualization. Photographs of several domain 6 drill holes were reviewed for quality control purposes.

Figure 16-6 shows the plan views of the data as visualized in Leapfrog.

Figure 16-6: Plan Views of the HW 10 M above the Mineralization in Drill Core, showing the Relative Length of Rock in the Three Defined Weathering Categories



The following observations are made:

- The percentage length of core described as weakly weathered KPS (Category 3) is 32%, but decreases to 26% when evaluating the HW irrespective of KPS.
- The percentage length of core described as separated rock KPS (Category 2) is 15%, but increases to 20% when evaluating the HW irrespective of KPS (including medium to strongly weathered non-KPS rock). Most of the increase is in Domain 2, whereas Domain 6 rock is less weathered.
- The percentage length of core described as disintegrated KPS (Category 1) is 3.5% when considering KPS siltstone only, and reduces to 2% when evaluating the HW irrespective of KPS. The decrease is observed in all domains.

Differences were observed in the most recent 2011 analysis due largely to the recognition of other rock types (besides KPS siltstone) in the HW to the mineralization. If the boundaries of the defined Resource change, the analysis will have to be repeated.

The criteria that SRK has used to determine potential problematic HW rock mass are based primarily on the amount of disintegrated rock in the HW to the mineralization, and on the density of data to support the analysis. The amount of separated (or strongly weathered) and weakly weathered rock is also a secondary consideration to the analysis.

16.2.9 Recommended Exclusion Zones

SRK recommends that the exclusion percentage values presented in Table 16-1 be used, based on the HW analysis and the new drill holes presented in Figure 16-6. A best-case and worst-case range is provided to reflect the uncertainty due to the wide-spacing of data and lack of understanding of the structures and associated alteration-deformation zones. The ranges can be used in a sensitivity analysis.

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 drill hole 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 study that mining is not considered in such areas.

Table 16-1: Recommended Exclusion Zone Percentage Ranges (April 2011)

Domain	Exclusion Zone %	Extraction ratios per depth (%)				Confidence Level
		<200m	200-400m	400-600m	600-800m	
1	30-40	78	75	70	60	Moderate
21	30-50	78	75	70	60	Low
22	15-20	78	75	70	60	Moderate
23	20-40	78	75	70	60	Moderate
3	20-30	78	75	70	60	Low
4	20-30	78	75	70	60	Low
5	20-30	78	75	70	60	Low
6	5-20	78	75	70	60	Moderate

16.3 Open Pit Mining

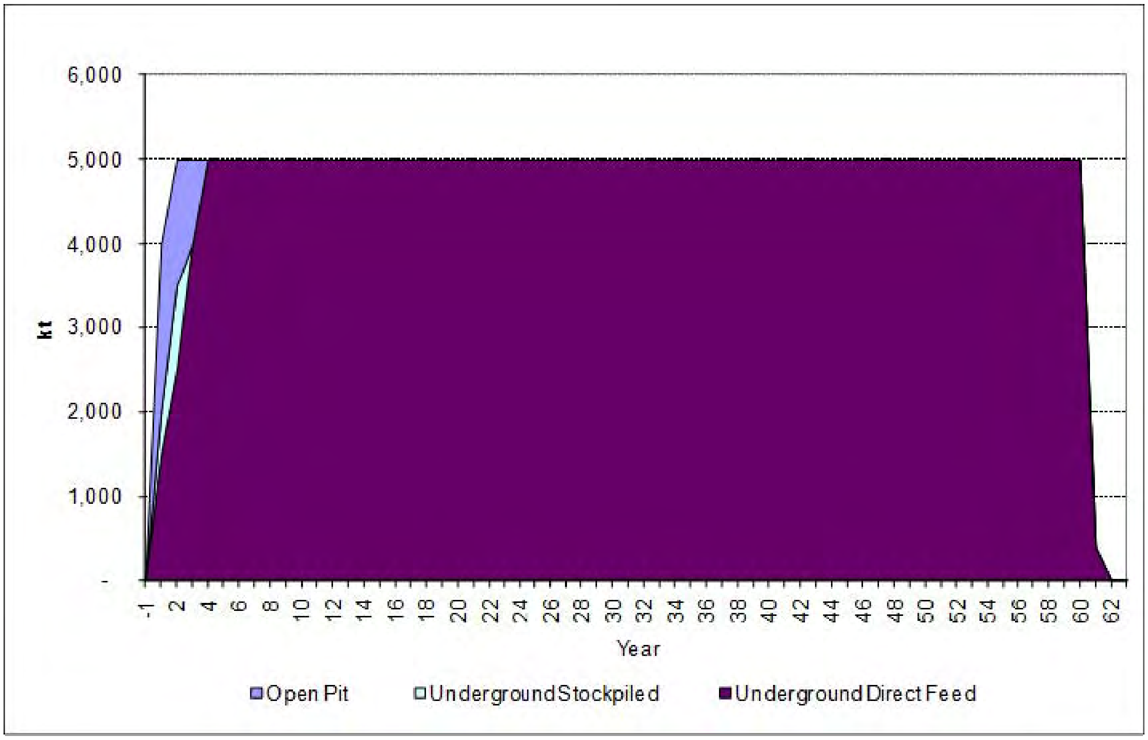
16.3.1 Introduction

The production schedule used to estimate costs and revenues for the PEA are based on a annual production rate of 5 Mtpa. The production schedule is based on mineral resources. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

In this scenario, an initial open pit is necessary to supplement the plant feed for the start-up of the processing plant whilst the underground mining is being established. It is expected that the underground mining will achieve the 5 Mtpa plant feed rate by Year 4, and will then be the sole plant feed source (Figure 16-7). The open pit produces a total of 4.5 Mt of plant feed over three years.

The open pit is scheduled to start at Year -1 (1 year before the processing plant), and will last for 4 years, with an initial total movement rate of 15 Mtpa (feed and waste combined), which decreases with the ramp-up of the underground production along the years. The plant start-up is planned for Year 1, with capacity to beneficiate 5 Mtpa.

Figure 16-7: Total Processed



Four different pits were created for this study and are explained in Section 16.3.2. The basic criteria utilized when designing the open pits for Kamoa Copper project are as follows:

- Truck to be used: CAT 785D (width: 7.1 m) or Komatsu HD1500-7 (width: 6.9 m)
- Haul road width: 25 m (3.5 x width of truck)
- Ramp gradient: 1:10
- Bench height: 10 m
- Berm width: 5 m
- Wall angle: 70°
- Berm on every bench: yes
- Maximum overall angle (top crest to bottom toe for a 50 m deep pit): 52.6°.

The preliminary open pit slope criteria were discussed and agreed with SRK. For the next stage of investigation the geotechnical criteria has to be confirmed by geotechnical drilling, logging and laboratory testing.

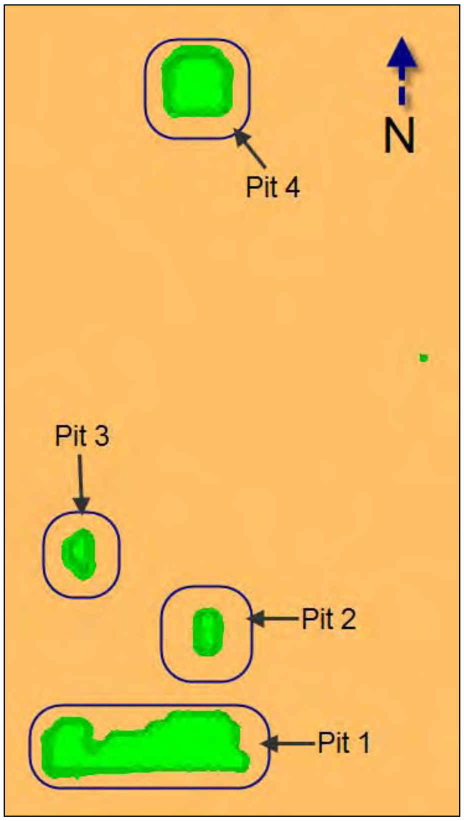
16.3.2 Open Pit Designs

The pits were designed targeting a total production rate of 5 Mtpa plant feed. Only Indicated resource was used for the pit optimization. The pit optimization results are shown in Table 16-2. The designs for these pits were based on Pit Shell 5. The four pit designs are shown in Figure 16-8.

Table 16-2: Pit Optimization Results

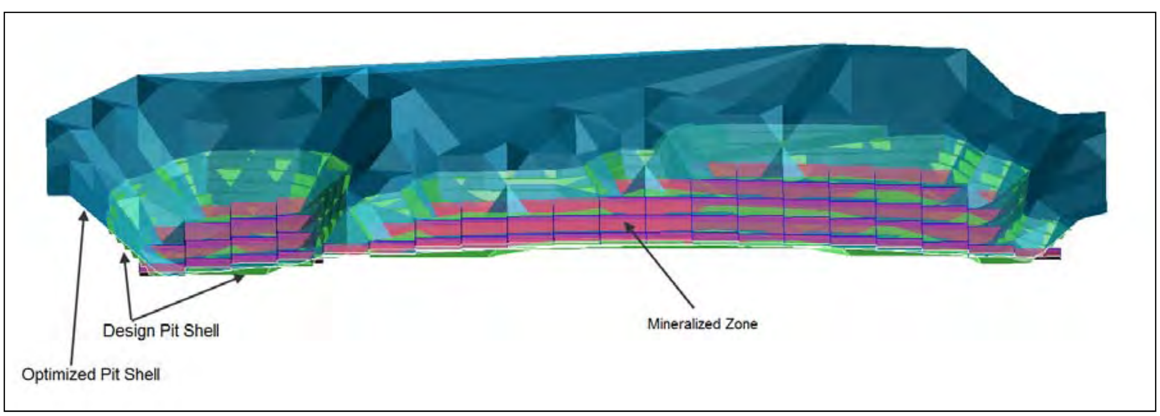
Pit Shell No.	Revenue Factor	Rock Mt	Stripping Ratio W:O	Production				Conc Mt	Rec Cu Mlb
				Feed Mt	Cu %	NSR US$/t Plant Feed	Rec %		
2	0.275	25	7.06	3.04	3.97	163	82	0.10	7.1
3	0.300	37	7.01	4.59	3.72	152	82	0.14	9.4
4	0.325	53	6.63	6.94	3.41	138	81	0.19	11.7
5	0.350	73	6.11	10.33	3.08	123	80	0.25	13.9
6	0.375	119	6.16	16.55	2.88	115	79	0.38	19.2
7	0.400	247	7.79	28.14	2.98	119	80	0.67	35.3
8	0.425	323	7.70	37.16	2.86	114	80	0.85	42.6
31	1.000	1,674	9.38	161.24	2.20	83	76	2.69	98.9

Figure 16-8: Pit Optimization 5 and Selected Pits



The designs ensured that the resource blocks contained in each individual optimized pit shell were fully included in the designed pit shells (Figure 16-9), and the bottom level of each designed pit was defined as the deepest point of each optimized pit shell.

Figure 16-9: Cross Section of Pit 1 and Pit Shell 5



Once the bottom level and the resource blocks to be contained into each pit were defined, the pits were then designed using the criteria outlined in Section 16.3.1.

The naming convention for the pit designs was defined as the word “Pit”, followed by a set of 2 numbers, where the first number is the number of the pit, and the second number is a provision for future cutback stages:

- **Pit 21:** Pit 2, original stage
- **Pit 22:** Pit 2, 1st cutback stage
- **Pit 23:** Pit 2, 2nd cutback stage
- **Pit 41:** Pit 4, original stage
- **Pit 42:** Pit 4, 1st cutback stage.

For this study, only the original stages of the pits were used.

The ramp exits, where possible, were designed towards north-east, where the plant and primary crusher are expected to be located. The pit designs can be seen in Figure 16-10 and Figure 16-11.

Figure 16-10:Pits 11, 21, and 31

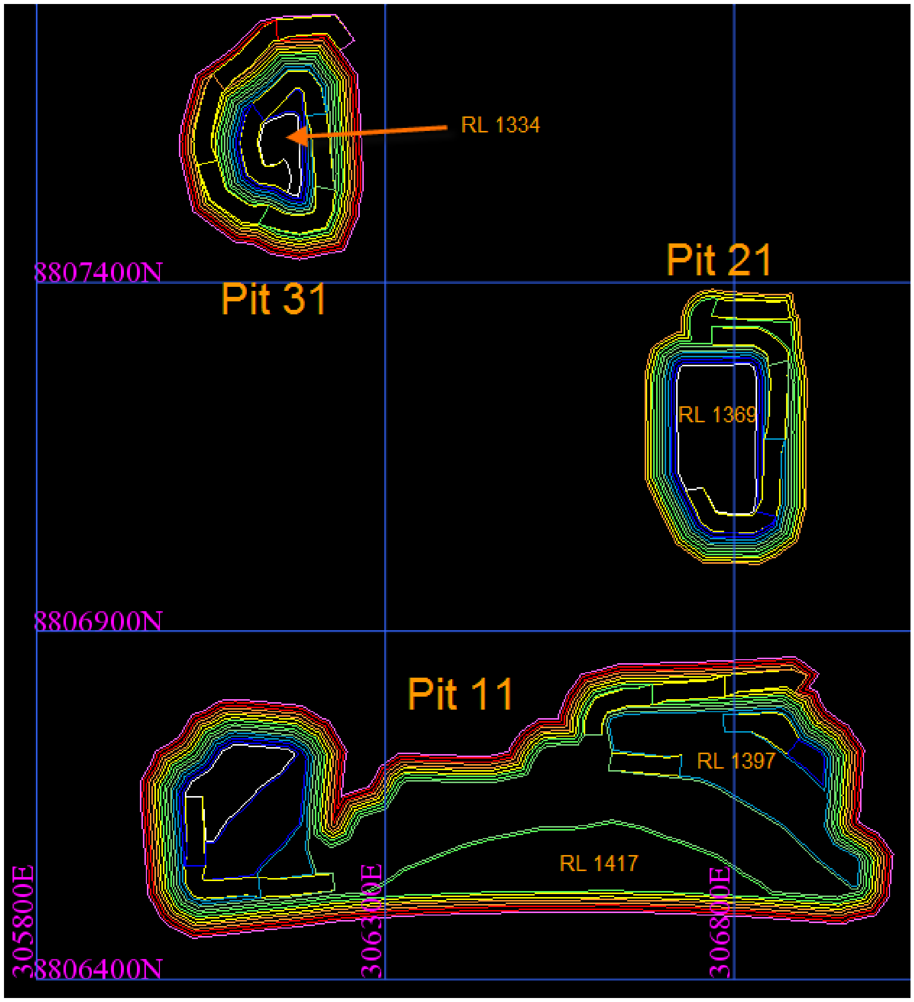
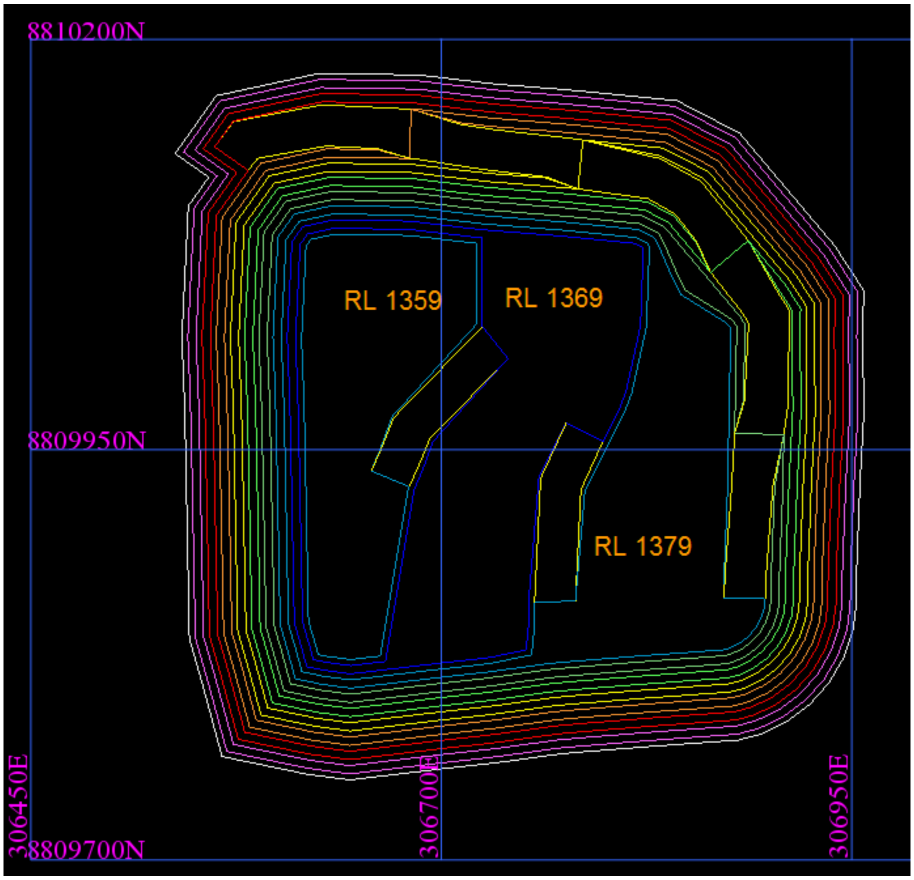


Figure 16-11:Pit 41



Note, that Pit 41 is the only one designed with the ramp exit in the north-west position. The reason for that is to avoid interaction of this ramp with eventual future cutbacks, maintaining the access to the bottom of the pit even after starting the cutbacks.

16.3.3 Open Pit Production Schedule

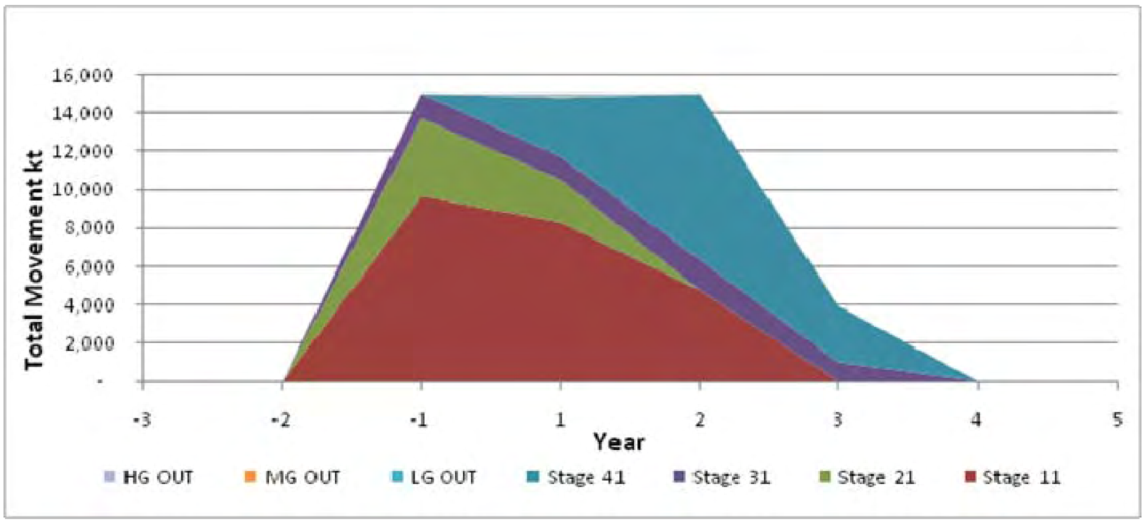
Year -1 is the first year of open pit mining, 15 Mt of total material is assumed to be mined from the open pit, only 0.2 Mt is plant feed, which will be stockpiled with the underground production. In the first year of underground mining (Year -2), only 500 kt of resource is expected to be mined from the underground mine. Along with an additional 1,000 kt of mining in Year -1, is assumed to be stockpiled to supplement plant feed in Year 1 when processing commences.

In Year 1 the plant starts to receive the stockpiled material, plus direct feed from open pit and underground production. Underground direct feed is expected to be 1.5 Mt, to which will be added 500 kt of stockpiled material from Year -2, 0.2 Mt of open pit stockpile from Year -1 and 1.8 Mt of direct feed from the open pit giving a total of 4 Mt of plant feed. The total open pit production rate is 15 Mtpa (including plant feed). Year

3 is the last year of open pit production. During this year, the overall open pit production drops from 15 Mt to 3.9 Mt, of which 1 Mt is plant feed. With 4 Mt from underground this results in a total of 5 Mt of plant feed.

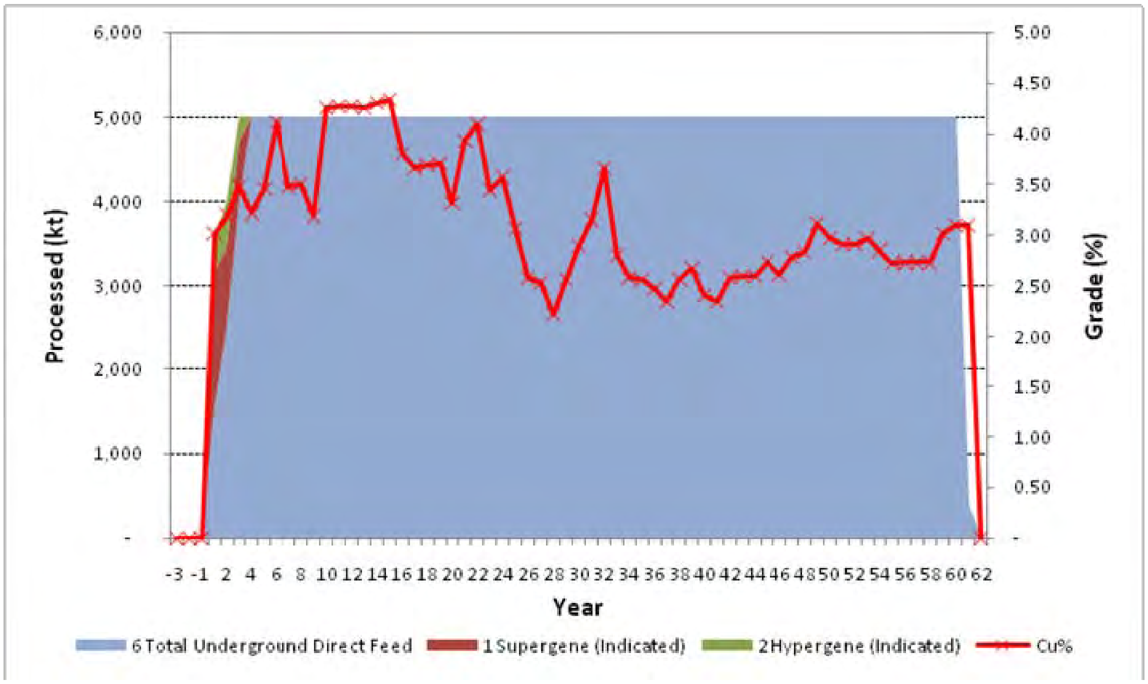
At Year 4, underground plant feed production is increased to the full production rate of 5 Mtpa. Open pit operations cease and all remaining open pit equipment is decommissioned. A summary of the 6 years of open pit production can be seen in Figure 16-12.

Figure 16-12: Total Open Pit Movement including Re-Handle



The underground production rate of 5 Mtpa is maintained until Year 60. The life-of-mine ends on Year 61, with only 388 kt of underground plant feed being mined during that year. The life-of-mine total feed processed by tonnage, grade and material type is shown in Figure 16-13.

Figure 16-13: Total Feed Processed and Grade by Material Type



16.4 Underground Mining

The mineral resources available for mining at the Project were based on a Datamine block model developed for the September 2011 Mineral Resource estimate and provided to Stantec in September 2011. The resources were classified as Indicated and Inferred based on the density of drilling information. The distribution of tonnes and grade by resource classification is shown in Table 16-3. These resources include the hypogene mineral as well as the supergene enrichment in the Kamoa Resource area.

Table 16-3: Distribution of Tonnes and Grade

	Enrichment	Mt	Total Cu (%)
Indicated	Supergene	119.7	2.12
	Hypogene	178.9	2.97
Inferred	Supergene	113.9	1.92
	Hypogene	362.6	2.79
Total	Supergene	233.7	2.02
	Hypogene	541.5	2.85
Total Tonnes		**775.1**	**2.60**

Figure 16-14 shows the graph of tonnes versus total Cu grade from the Kamoa mineral resource model and includes all indicated and inferred resources within the limits of the September 2011 model. Given the large extent of the mineralized zone, a preliminary cut-off grade of 2% total Cu was used to assess the available mineral resource.

Figure 16-14:Kamoa Available Resources (Tonnage versus Copper Grade)

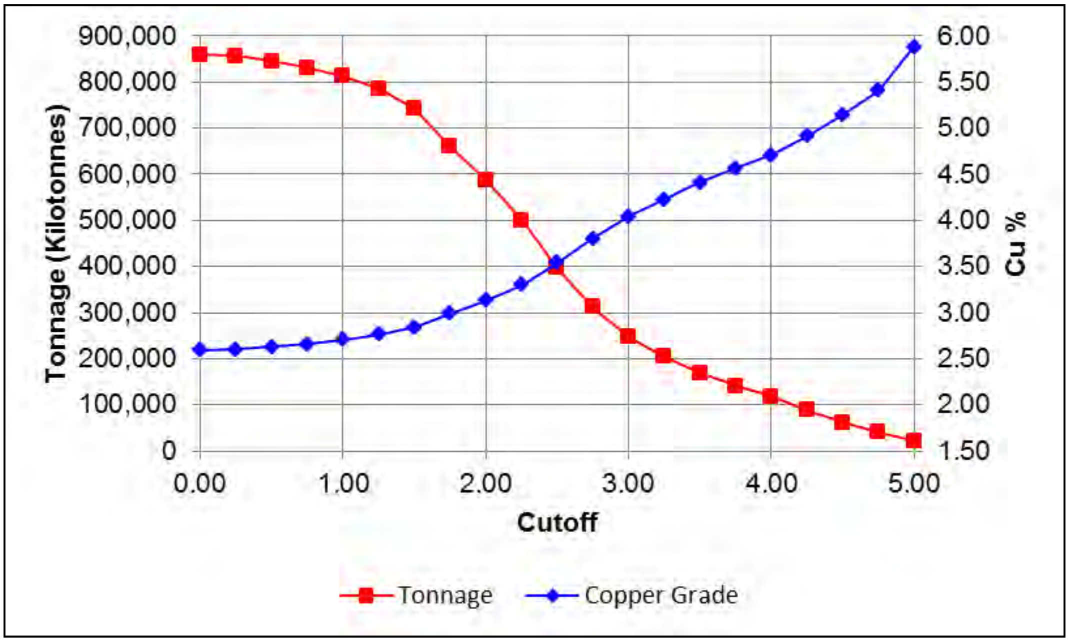
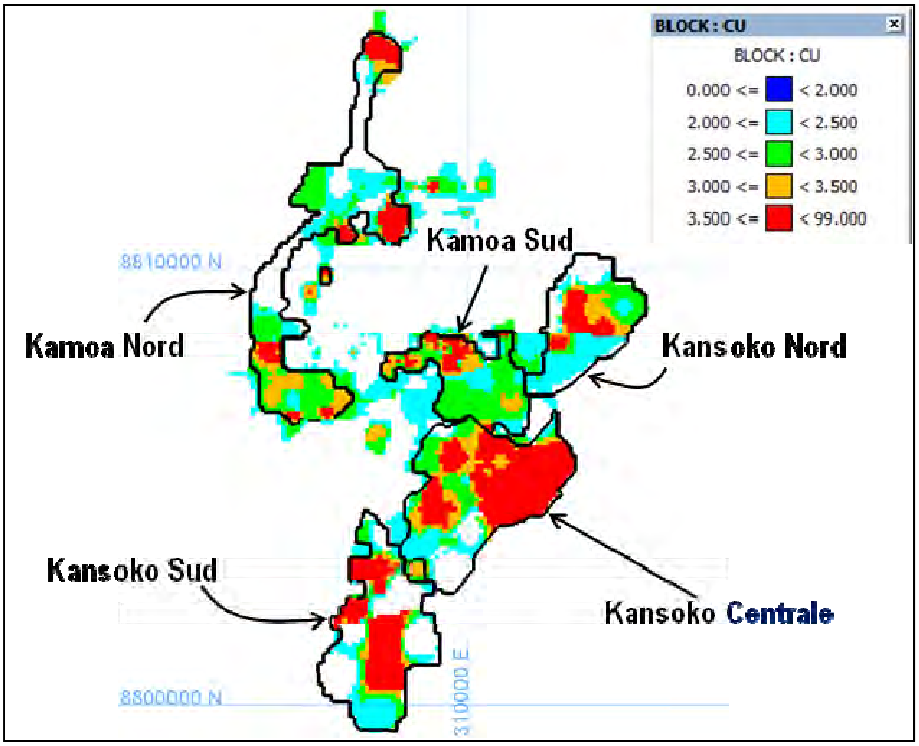


Figure 16-15 presents a plan view of the Kamoa Project showing the resources available at the cut-off value of 2% total Cu and the different sections identified for mining the Kamoa Resource.

Figure 16-15:Kamoa Resources at 2% Cu Cut-Off (showing Mining Sections)



16.4.1 Underground Dilution

Identification of the blocks for mining the Kamoa Resource was accomplished using the Vulcan Envisage three-dimensional (3-D) underground design software package. Apart from total Cu grade, other variables recorded in the block model included the thickness of the mineralization zone (thick), the depth below surface (dist2surf) and the distance (dist2KPS) to the contact between the pyritic shale (KPS) and the diamictite. A minimum mining thickness of 3.5 m was used for this study. Blocks with distance to KPS less than 3.5 m were avoided to reduce inventory which could require higher ground support costs. Isolated blocks meeting the above criteria were not included within the diluted resource.

Appropriate recovery and dilution factors were used to generate the diluted blocks as discussed below.

The low dip and the flat, plate-like structure of the resource make it conducive to mining using the room-and-pillar mining method in the shallow portions of the mineralized zone, transitioning to a drift-and-fill mining method in the deeper sections of the mineralized zone. Room-and-pillar panels are designed to be 80 m wide and 500 m long with in-panel extraction ratios ranging from 60% to 80% depending on the panel depth below surface. Partial extraction of the barrier pillars (up to 50%) is planned at the end of mining of each section. The overall extraction ratio in the room-and-pillar areas is expected to be between 56–69% depending on the depth below surface. Higher in-panel extraction ratios (up to 95%) are expected within the drift-and-fill areas with an overall extraction ratio of 78% after partial extraction of barrier pillars.

In-stope dilution was estimated to be 0.3 m from the roof and 0.3 m from the floor and was calculated for each panel based on the thickness of the resource within the panel. The average dilution was estimated to be about 10%. The dilution values will depend, amongst other things, on the HW stability and this has to be confirmed by the underground exposure. If HW stability is worse than currently anticipated, for example due to fracturing and joint continuity, the dilution value will be most certainly higher.

Mining panels were laid out within each section (as shown in Figure 16-16) to generate diluted blocks. The diluted blocks by mining section are shown in Table 16-4.

Figure 16-16: Mining Panel Layout

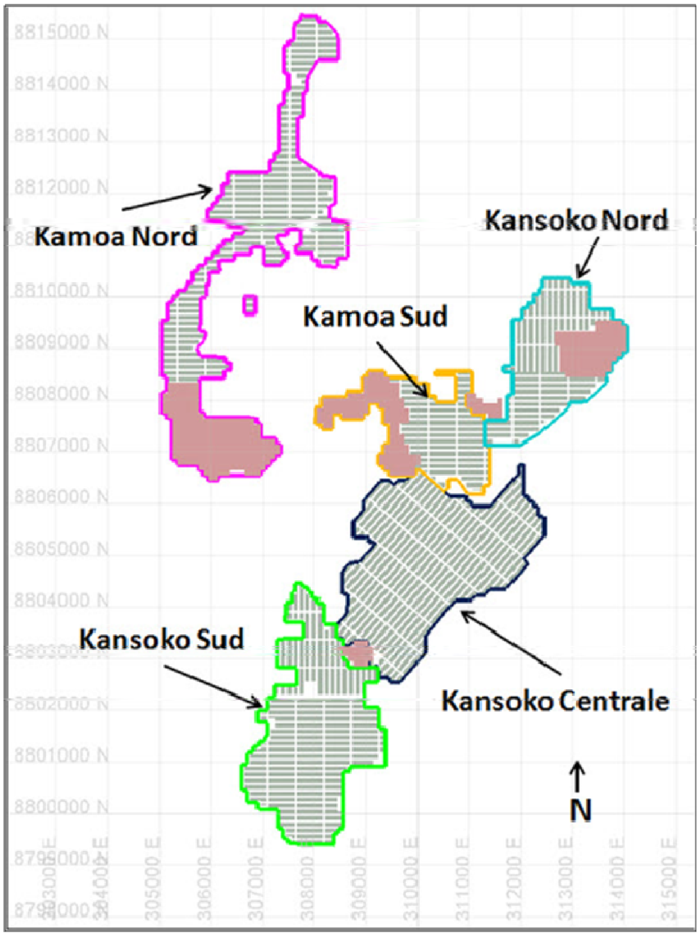


Table 16-4: Diluted Blocks

Section	Tonnes	Cu %
Kansoko Centrale	149,845,650	3.62
Kansoko Sud	105,254,698	2.86
Kansoko Nord	86,572,828	2.76
Kamoa Sud	56,347,308	2.91
Kamoa Nord	84,058,140	2.66
Total	**482,078,624**	**3.05**

16.4.2 Underground Production Schedules

The production schedule for mining of the Kamoa Resource were developed using the resources identified in Figure 16-17. The operating criteria used are as listed below.

- 360 operating days per year.
- Production includes all plant feed development and stope production.
- Production represents recovered/diluted plant feed presented on an annual basis.

The layout of the Room-and-Pillar panels in this flat-dipping mineralization zone results in early plant feed production because the panel access drifts will be developed within the mineralized zone.

At 5 Mtpa, the recovered/diluted plant feed at the Kamoa Project is mined over 63 years including 2 years of preproduction development. Mining is commenced in Kansoko Centrale and in Kansoko Sud within the indicated hypogene resources and slowly spread out to other sections of the resource.

Table 16-5 and Figure 16-17 present a summarised annual production schedule.

The schedule of mining is presented by resource category in Figure 16-18 and by type of enrichment in Figure 16-19.

Table 16-5: Underground Mine Production Schedule

Year	Plant Feed Tonnes	Cu (%)		Year	Plant Feed Tonnes	Cu (%)
-2	500,000	3.60		31	5,000,000	3.15
-1	1,000,000	2.80		32	5,000,000	3.66
1	1,500,000	3.41		33	5,000,000	2.80
2	2,500,000	3.47		34	5,000,000	2.58
3	4,000,000	3.44		35	5,000,000	2.56
4	5,000,000	3.22		36	5,000,000	2.48
5	5,000,000	3.46		37	5,000,000	2.35
6	5,000,000	4.12		38	5,000,000	2.57
7	5,000,000	3.48		39	5,000,000	2.67
8	5,000,000	3.51		40	5,000,000	2.41
9	5,000,000	3.19		41	5,000,000	2.35
10	5,000,000	4.26		42	5,000,000	2.59
11	5,000,000	4.28		43	5,000,000	2.59
12	5,000,000	4.27		44	5,000,000	2.60
13	5,000,000	4.26		45	5,000,000	2.74
14	5,000,000	4.31		46	5,000,000	2.61
15	5,000,000	4.34		47	5,000,000	2.79
16	5,000,000	3.80		48	5,000,000	2.83
17	5,000,000	3.66		49	5,000,000	3.11
18	5,000,000	3.70		50	5,000,000	2.97
19	5,000,000	3.71		51	5,000,000	2.91
20	5,000,000	3.40		52	5,000,000	2.91
21	5,000,000	3.93		53	5,000,000	2.97
22	5,000,000	4.10		54	5,000,000	2.86
23	5,000,000	3.45		55	5,000,000	2.73
24	5,000,000	3.58		56	5,000,000	2.73
25	5,000,000	3.06		57	5,000,000	2.73
26	5,000,000	2.58		58	5,000,000	2.73
27	5,000,000	2.53		59	5,000,000	3.02
28	5,000,000	2.22		60	5,000,000	3.10
29	5,000,000	2.57		61	388,025	3.10
30	5,000,000	2.90		Total	294,888,025	3.15

Figure 16-17: Underground Production Schedule

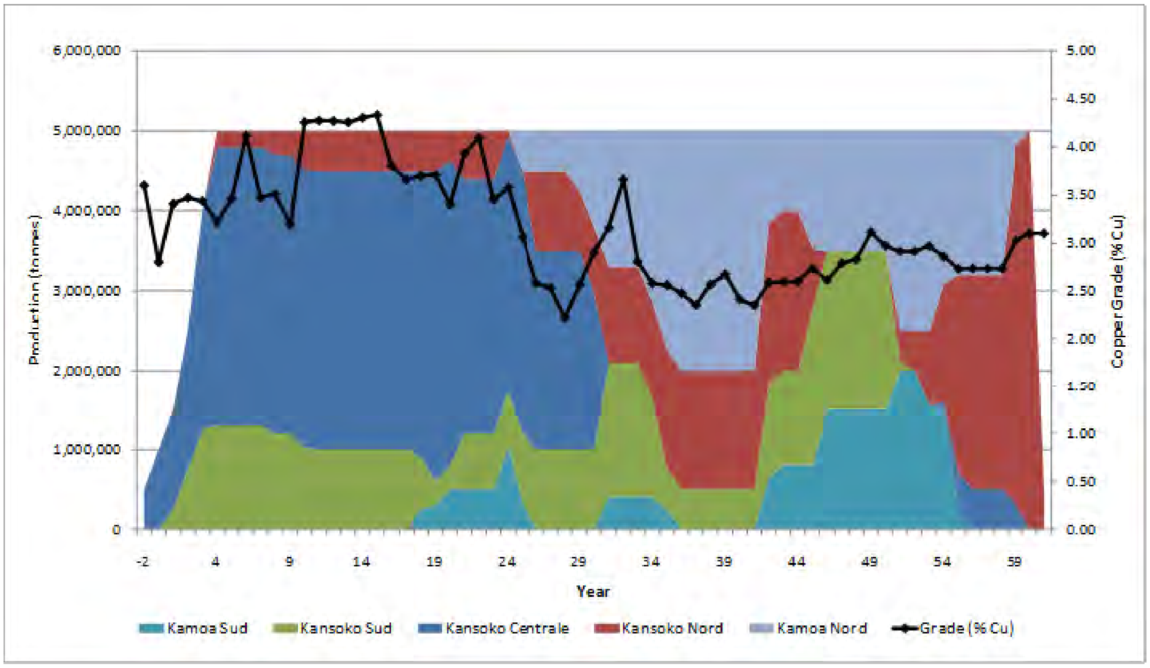


Figure 16-18: Underground Mining Schedule, by Resource Category

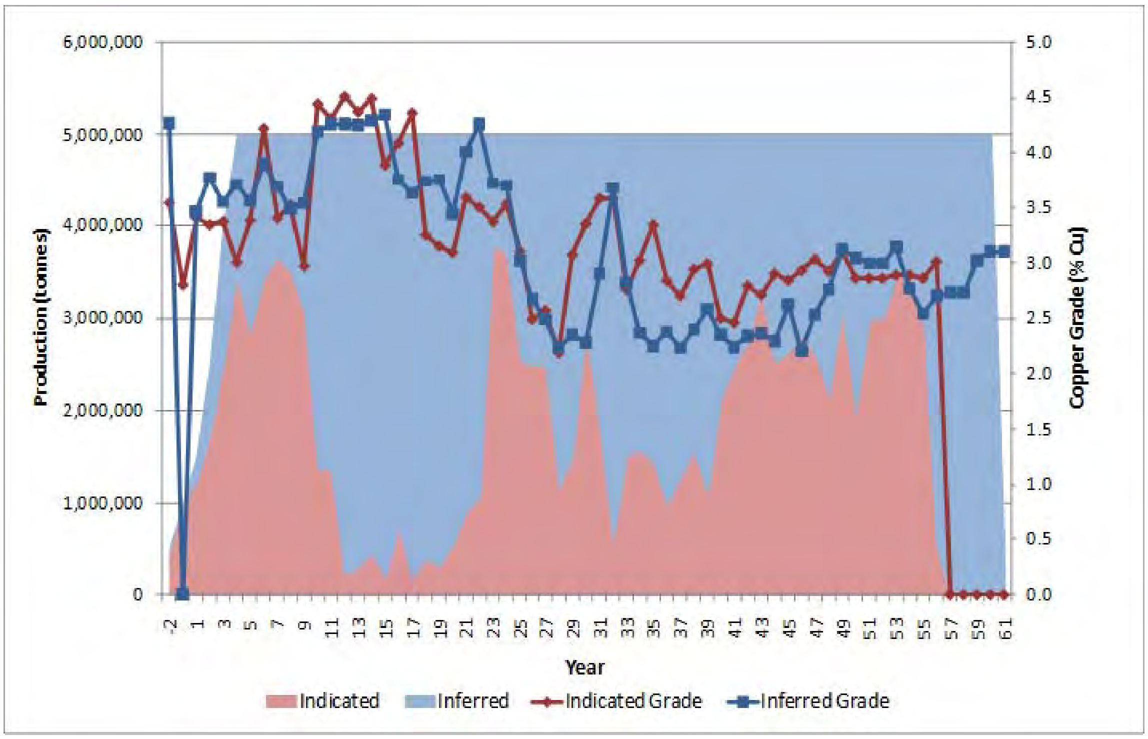
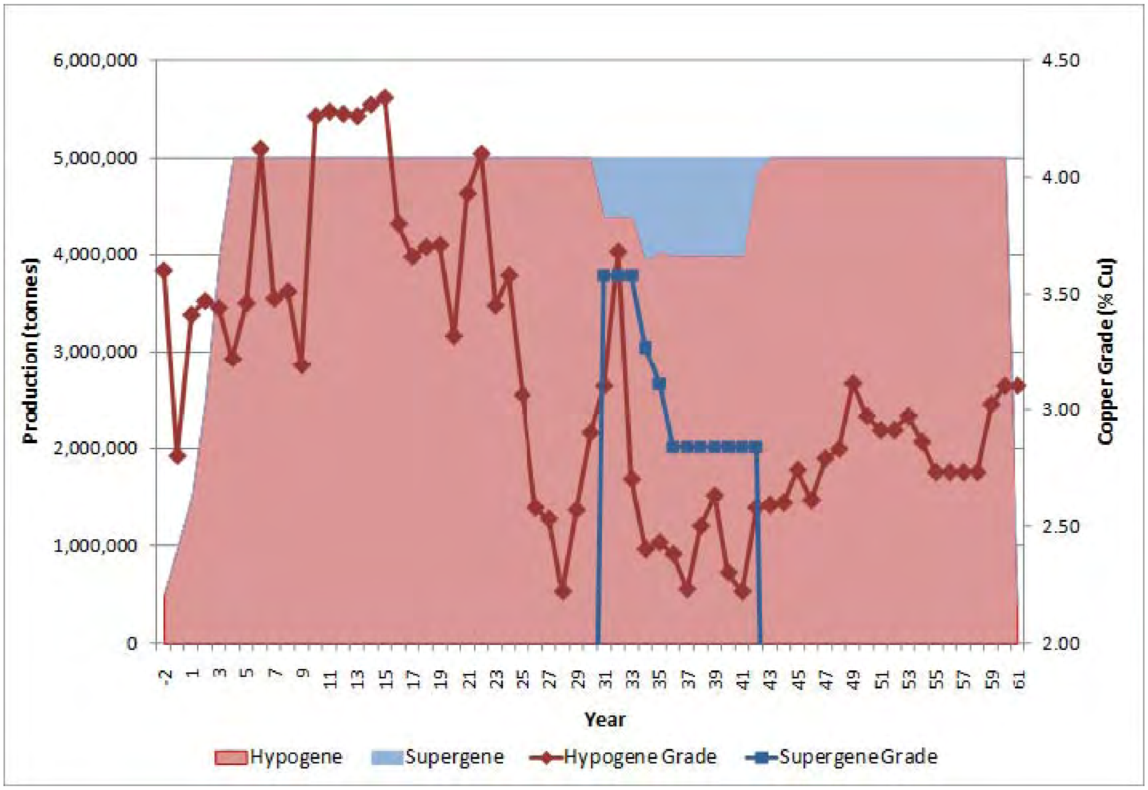


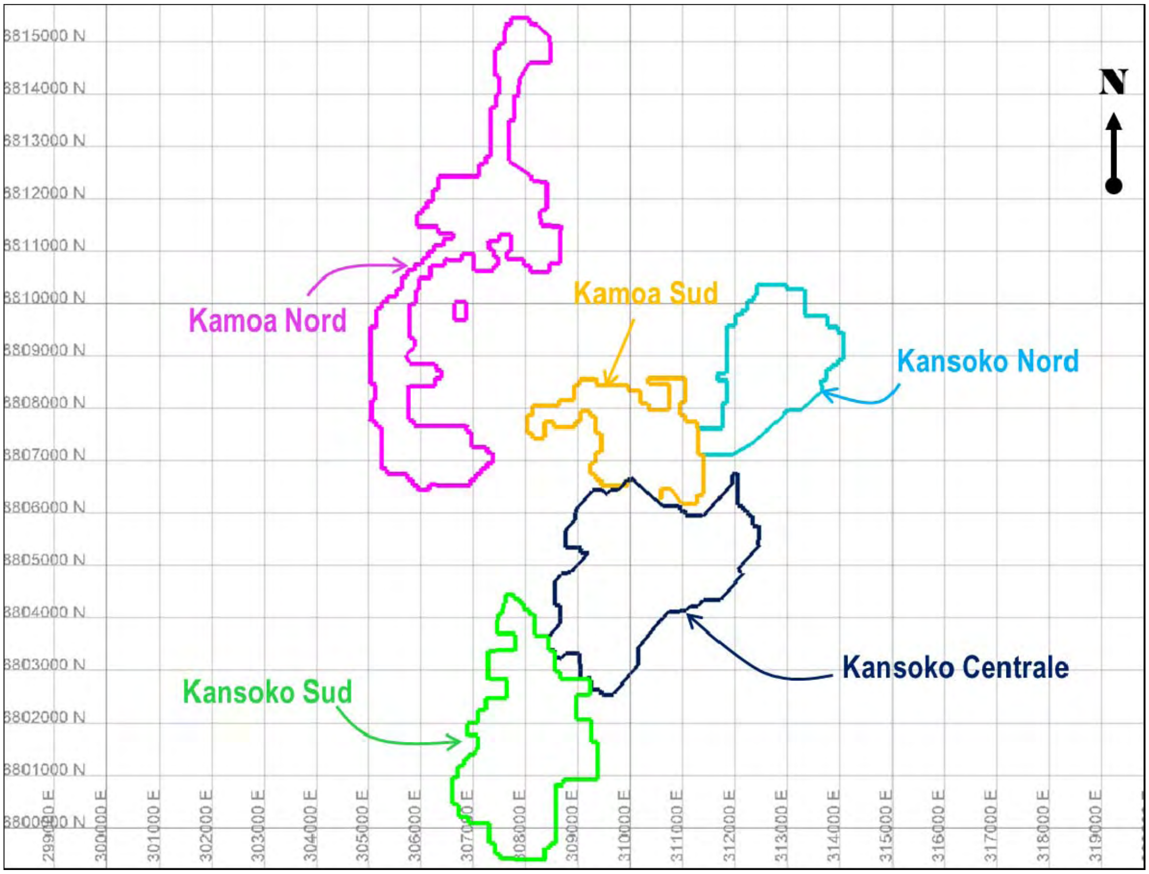
Figure 16-19:Underground Mining Schedule, by Type of Enrichment



The mine design for the Kamoa Project is based on a total resource model of 295 million tonnes and achieving a 5 Mtpa production rate from mining in multiple sections of the mineralized zone. The design incorporates early access to the Kansoko Centrale section, while developing additional accesses to Kansoko Sud, Kansoko Nord, and Kamoa Nord sections. Access to the Kamoa Sud is through the accesses developed for Kansoko Nord. The accesses to each section include a main access decline and a conveyor decline. Additional infrastructure requirements such as surface and underground offices, surface and underground maintenance facilities, ventilation raises, and cemented rock fill (CRF) plants are designed to support operations at the 5 Mtpa level.

Figure 16-20 shows an overview of the Kamoa Resource area. Since the Kansoko Sud, Kasoko Nord, and Kamoa Nord sections are at considerable distances from the main Kansoko Centrale, separate infrastructure will be required for each of these sections. Kansoko Centrale is the largest single resource section with over 33% of the total Kamoa production. The cost estimates for the designs developed for the Kansoko Centrale were scaled to the production in the other sections. The production schedule is based on mineral resources. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

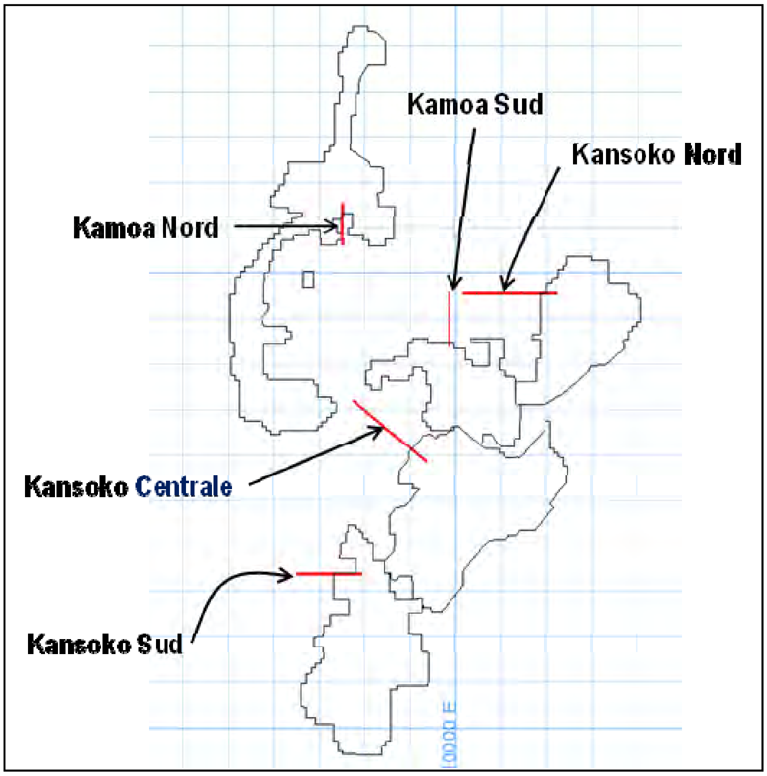
Figure 16-20: Overview of Kamoa Resource Area (showing Location of Mining Sections)



Kansoko Centrale

Kansoko Centrale constitutes about 33% of the known Kamoa Resource as at September 2011, and has been drilled extensively from the surface. The section is accessed through a main access decline and a conveyor decline, both designed at -8.5 degrees (15%) gradient from surface. Development of the accesses for this section commences simultaneously as those for Kansoko Centrale and in a similar fashion. However, the primary access drifts in this section are laid out along azimuth 127, ensuring the stoping panels can be laid out at an apparent dip of 8.5 degrees (15%). These declines are directed toward the highest grade portion of the section as shown in Figure 16-21.

Figure 16-21:Development of Declines



Once the access declines reach the mineralized zone, two parallel primary access drifts (5 m high by 5 m wide) are developed within the mineralization zone along azimuth 131, which allows for mining of the panels along an apparent dip of 3.8 degrees (6.7%) as shown in Figure 16-22. Stoping panels (80 m wide by 500 m long) are laid out on either side of the primary access drifts with 40 m wide pillars between adjacent panels. Additional drifts are developed parallel to the primary access drifts so that a checkerboard of stoping panels is laid out within the section. Accesses to the stoping panels are developed from these primary access drifts. The layout of panels within the Kansoko Centrale is shown in Figure 16-23.

All the drifts are developed in mineralization with adequate ground support to ensure their long term stability. Intersections with accesses to the stoping panels may be heightened and provided with additional ground support with cable bolts, to facilitate the loading of the panel conveyors by LHDs operating within the stoping panels. These drifts also serve as ventilation airways to provide adequate ventilation to the stoping panels.

Figure 16-22: Mining Room and Pillar Stopes along Apparent Dip

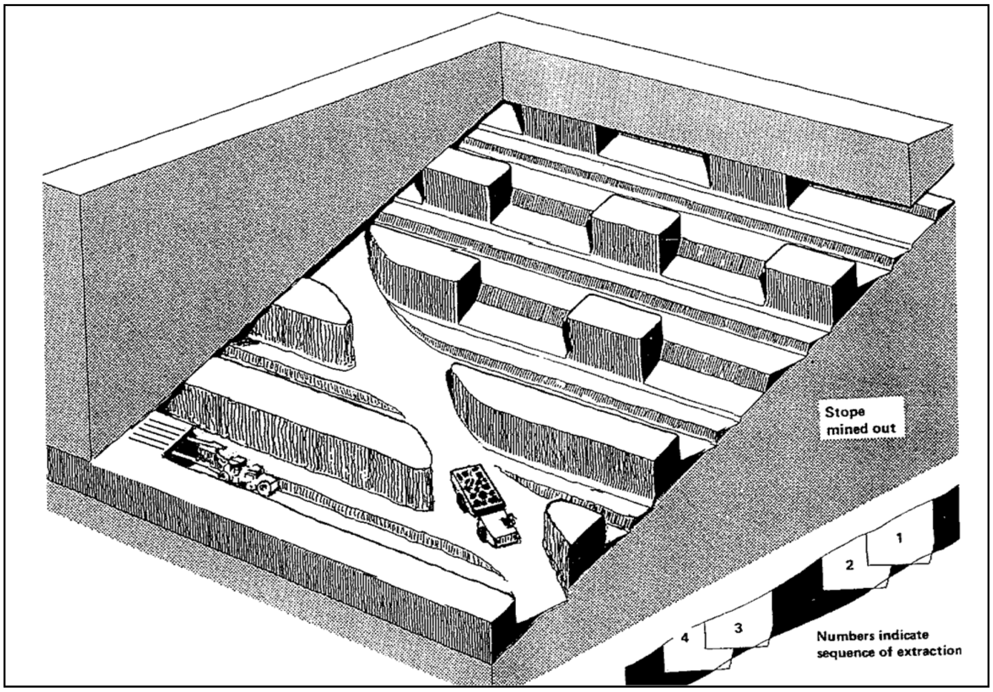
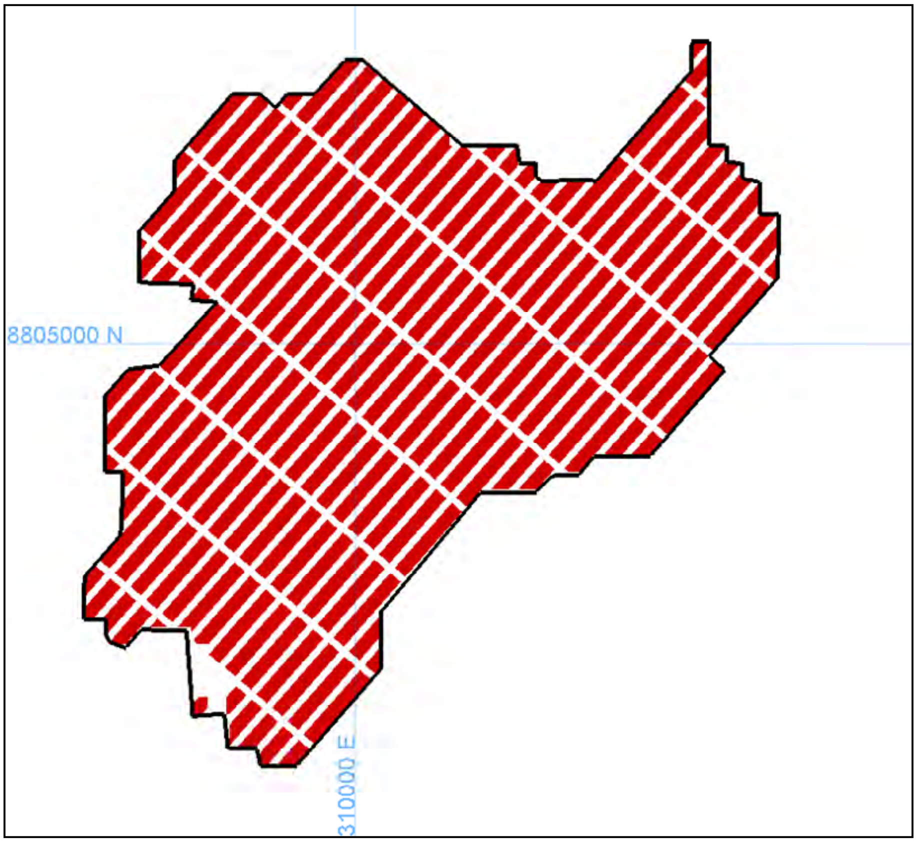


Figure 16-23:Layout of Stoping Panels in Kansoko Centrale



Approximately 15 million cubic metres of CRF will require to be placed within the drift and fill sections of Kansoko Centrale at a maximum rate of 1.3 million cubic metres per year. Provisions for the crushed rock required for this quantity of CRF will be necessary as part of the overall plan for the development of the Kamoa project.

Infrastructure required to support the operations within the Kansoko Centrale Section are designed based on the maximum production rate of 3.9 Mtpa. Since 42% of the Kansoko Centrale production lies at depths greater than 800 m, an underground CRF plant (providing the required rate of CRF) is included in the infrastructure requirements for the section. The design parameters for infrastructure development are presented in Section 18.

Kansoko Sud

Development of the accesses for this section commences simultaneously as those for Kansoko Centrale and in a similar fashion. However, the primary access drifts in this section are laid out along azimuth 127, ensuring the stoping panels can be laid out at an apparent dip of 8.5 degrees (15%).

The production in the Kansoko Sud Section (including Sections Sud A and Sud B) is about 54% of the resource in Kansoko Centrale. The infrastructure required for this section has been adjusted to reflect the smaller resource. Cost estimates for the infrastructure were reduced accordingly for the mining of this section. Since 7% of the Kansoko Sud production lies at depths greater than 800 metres, an underground CRF plant (providing the required rate of CRF) is included in the infrastructure requirements for the section.

Approximately 1.4 million cubic metres of CRF will require to be placed within the drift-and-fill sections of Kansoko Centrale at a maximum rate of 0.67 million cubic metres per year. Provisions for the crushed rock required for this quantity of CRF will be necessary as part of the overall plan for the development of the Kamoa project.

Kansoko Nord

Development of the accesses for this section commences four years after the start of production from Kansoko Centrale and in a similar fashion. The primary access drifts in this section are laid out along azimuth 160, ensuring the stoping panels can be laid out at an apparent dip of 8.5 degrees (15%).

The production in the Kansoko Nord Section (including Sections Nord A and Nord B) is about 60% of the September 2011 Mineral Resource estimate in Kansoko Centrale. The infrastructure required for this section has been adjusted to reflect the smaller resource. Cost estimates for the infrastructure were reduced accordingly for the mining of this section. Since 18% of the Kansoko Nord production lies at depths greater than 800 m, an underground CRF plant (providing the required rate of CRF) is included in the infrastructure requirements for the section.

Approximately 3.8 million cubic metres of CRF will require to be placed within the drift-and-fill sections of Kansoko Nord at a maximum rate of 0.6 million cubic metres per year. Provisions for the crushed rock required for this quantity of CRF will be necessary as part of the overall plan for the development of the Kamoa project.

Kamoa Sud

Production from this section commences 18 years after the start of production from Kansoko Centrale. The access for this section is developed from the main accesses developed for Kansoko Nord. The primary access drifts in this section are laid out alongazimuth 160, ensuring the stoping panels can be laid out at an apparent dip of 8.5 degrees (15%).

The production in the Kamoa Sud Section is about 24% of the estimated September 2011 Mineral Resource in Kansoko Centrale. The infrastructure required for this section has been adjusted to reflect the smaller resource. Cost estimates for the infrastructure were reduced accordingly for the mining of this section. All of the Kamoa Sud production lies at depths less than 800 m and no underground CRF plant is required for the section.

Kamoa Nord

Development of the accesses for this section is commenced 25 years after the start of production from Kansoko Centrale and in a similar fashion. This section is currently estimated to be flat dipping (dip less than 5 degrees) and the primary access drifts in this section can be laid out in any direction.

The production in the Kamoa Nord Section is about 60% of the September 2011 Mineral Resource estimate in Kansoko Centrale. The infrastructure required for this section has been adjusted to reflect the smaller resource. Cost estimates for the infrastructure have been reduced accordingly for the mining of this section. All of the Kamoa Nord production lies at depths less than 800 m and no underground CRF plant is required for the section.

16.4.3 Underground Mine Production Rates

The production rates for different sections of the September 2011 Kamoa Resource are based on achieving an overall production rate of 5 Mtpa. Based on the resources available, 5 Mtpa equates to approximately 61 years of plant feed production. The pre-production period is approximately 2 years for a total project life of 63 years. Since the primary access drifts in each section are driven in mineralized zones, the development tonnes contribute to the production schedule.

The design also allows multiple mining areas to be opened up concurrently, reducing the time required to achieve the target daily production rate. The combined plant feed production rate is 5 Mtpa, which equates to 13,889 tpd, assuming 360 operating days per year.

16.4.4 Mining Method

Given the favourable characteristics of the September 2011 Mineral Resources, including its relatively undeformed, continuous mineralization, local continuity demonstrated by close-spaced drilling and flat to moderate dips, it is considered amenable to large-scale, room-and-pillar mining mechanized mining, as well as drift-and-fill mining. The principal mining method is room-and-pillar for Mineral Resources to 800 m below surface and drift-and-fill for Mineral Resources below that level, as shown below. These conventional mining methods are widely used across the industry.

The philosophy of the current design is based on the provision of stable pillars (with factor of safety greater than 1.2) within the stoping panels in each section. SRK Consulting updated the guidelines in Table 16-1 based on better estimates of UCS and recommended extraction ratios for different mineralization thicknesses and depths below surface as shown in Table 16-6.

Table 16-6: Guidelines for Extraction Ratios Based on Mineralization Thickness and Depth below Surface

Depth below Surface	Mineralization Thickness (m)	UCS (MPa)	Primary Recovery (%)	Total Recovery (%)	Factor of Safety
200 m	4	90	82	90	1.7
	8	90	75	85	1.5
350 m	6	90	71	80	1.5
	8	90	65	75	1.6
550 m	5	90	60	65	1.5
	7	90	54	60	1.5
800 m	5	90	48	48	1.5
	8	90	38	38	1.5

These guidelines were used to develop room-and-pillar sizes within the stoping panels as shown in Table 16-7. The layout within a room-and-pillar stoping panel for depth less than 350 m is presented in Figure 16-24 and the layout for a panel for depths between 350–800 m is shown in Figure 16-25.

Table 16-7: Room and Pillar Sizes within Stoping Panels

Depth below Surface	Mineralization Thickness (m)	Room Width (m)	Pillar Width (m)	Total Recovery (%)	Factor of Safety
<350 m	6	10	8	80	1.5
	8	10	10	75	1.6
350-800 m	5	8	12	64	1.5
	7	8	14	60	1.5
>800 m	5	8	8	95	Mined as Drift-and-Fill
	8	8	8	95	

Figure 16-24: Typical Plan Layout of Room and Pillar Stoping Panel (for depth less than 350 m)

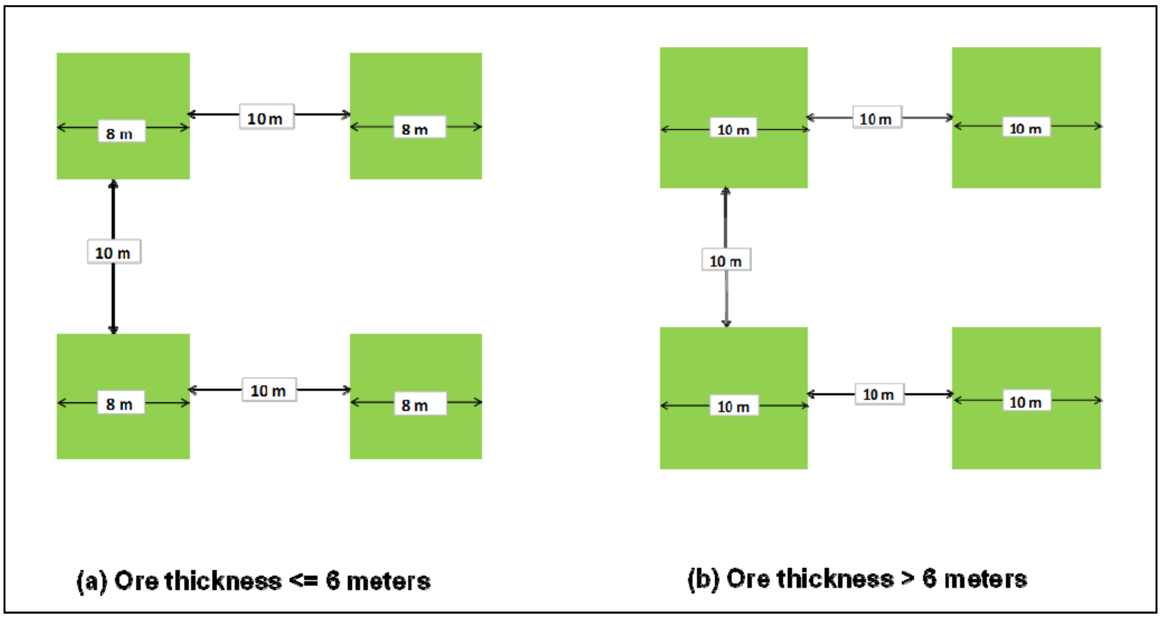
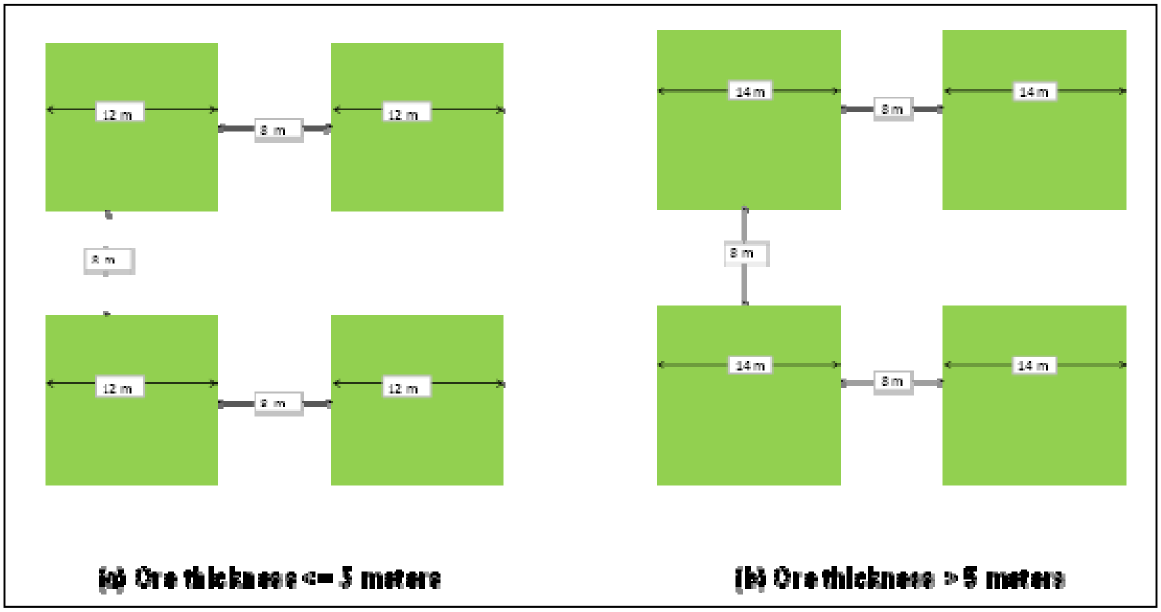
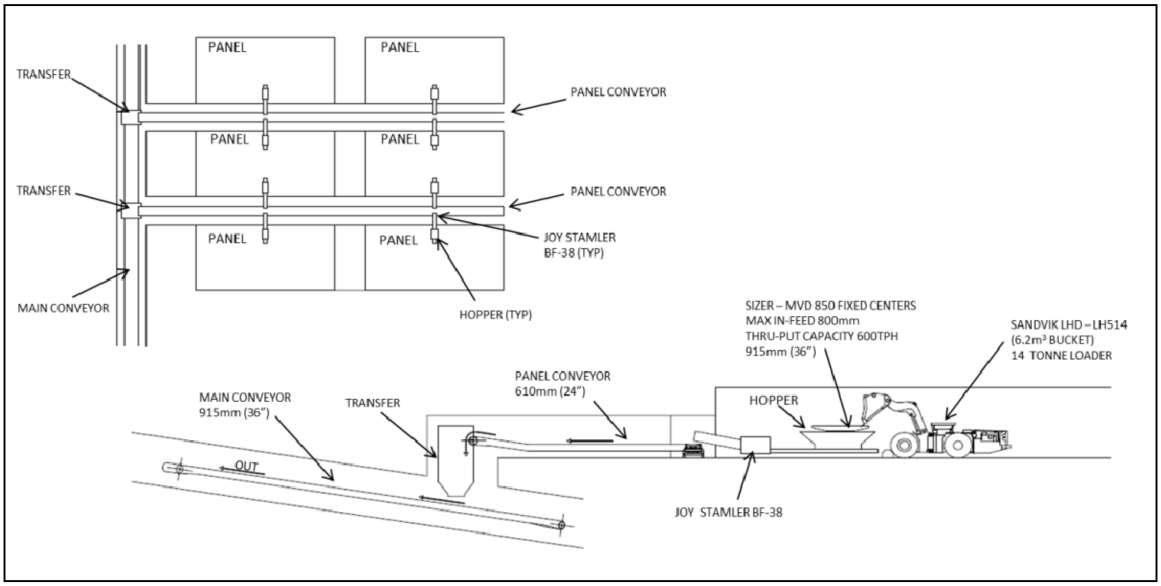


Figure 16-25: Typical Plan Layout of Room and Pillar Stoping Panel (for depths between 350-800 m)



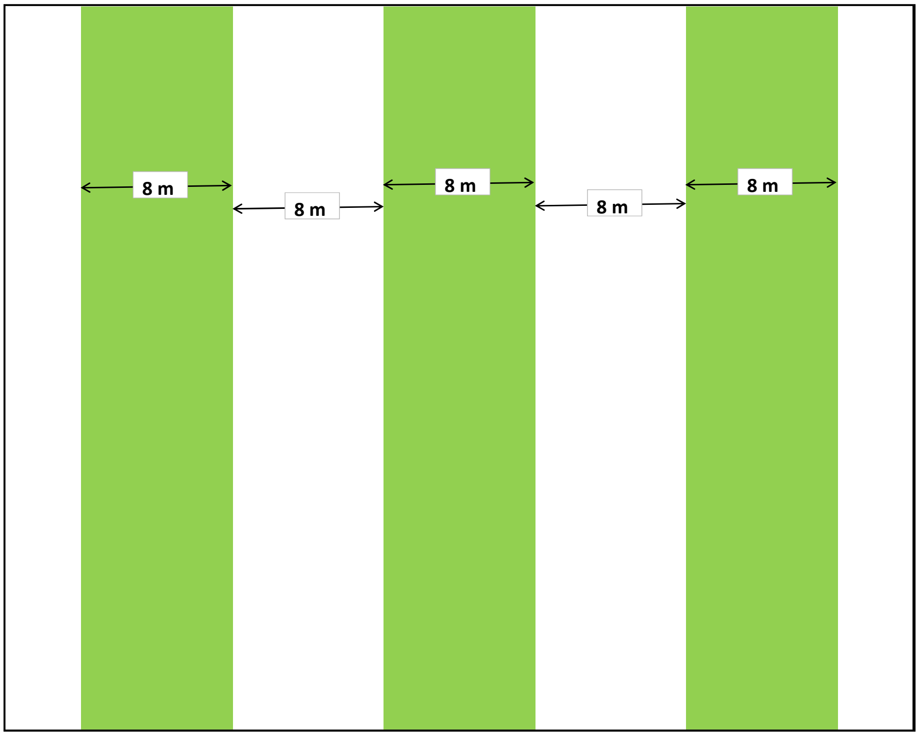
Each panel is developed with a 5 m wide drift using large three-boom drill jumbos. The initial drift is then slashed out to the maximum room size based on Table 16-7, leaving regular square pillars as per design. The muck from the panel is hauled by an LHD to an intersection in the primary access drift, where it is dumped into the hopper of a feeder-breaker. The muck is crushed to less than 600 mm size and fed onto a 24-inch panel conveyor, which leads the material to a transfer station to be loaded onto the main 36-inch conveyor in the conveyor decline, and thence to a stockpile near the portal. The plant feed can then be transported to the mill from the stockpile. Figure 16-26 shows the material flow from the panels to the surface stockpile.

Figure 16-26:Material Flow from Stoping Panels to Surface



For panels greater than 800 m depth below surface, Drift-and-Fill mining is employed to provide additional support to the working areas. Drifts are mined to an 8 m width from one end of the panel and then backfilled. A primary-secondary sequence of mining is followed with all primary stopes being backfilled with CRF and stabilized within the panel before the mining of the intervening secondary stopes. A typical layout for Drift-and-Fill panels is shown in Figure 16-27.

Figure 16-27: Typical Layout of Drift and Fill Stoping Panel



The pillars between the stoping panels are extracted after completion of mining within the section. The pillar extraction will be completed on a retreat mining sequence ensuring proper access and ventilation is maintained to the active mining areas.

Mine Design Parameters

The design parameters for the mining areas in Kamoa consider high-level geotechnical information provided by SRK Consulting. The geotechnical information is utilized for determining appropriate mining methods and the dimensions for the drifts and pillars as shown in Table 16-7. It has to be understood that such criteria are very preliminary and are applicable only to very competent ground conditions areas. There is considerable uncertainty about the structural character of the deposit. The current rock mass characterization assumptions, pillar and hanging wall stability and panel scale orebody geometry has to be confirmed by the underground exposures. The achievable spans for both room-and-pillar, and drift-and-fill methods have to be confirmed by the numerical modelling analyses.

Each section of the Kamoa Project is accessed through a main access decline and a conveyor decline. Additional waste development is required in the form of ventilation raises and transfer points. Since all sections in the Kamoa Project are developed within the mineralized zone, other infrastructure requirements are met largely through excavations developed in mineralized zones. The infrastructure design parameters for excavations developed within the waste rock for the Kamoa Project are presented in Table 16-8 for each section. Allowances are made within the estimates for muck bays and cross-overs from the two parallel declines.

Table 16-8: Infrastructure Design Concepts in Waste for Kamoa Project

Section		Height (m)	Width (m)	Total Length (m)	Comments
Kansoko Centrale	**Lateral Development**				
	Conveyor Decline	6.5	5.5	2,200	–
	Main Access Decline	6.5	5.5	2,200	–
	Vertical Development				
	Ventilation Raise	–	–	1,069	3 m diameter
Kansoko Sud	**Lateral Development**				
	Conveyor Decline	6.5	5.5	1,400	–
	Main Access Decline	6.5	5.5	1,400	–
	Vertical Development				
	Ventilation Raise	–	–	837	3 m diameter
Kansoko Nord	**Lateral Development**				
	Conveyor Decline	6.5	5.5	2,200	–
	Main Access Decline	6.5	5.5	2,200	–
	Vertical Development				
	Ventilation Raise	–	–	896	3 m diameter
Kamoa Sud	**Lateral Development**				
	Conveyor Decline	6.5	5.5	1,300	–
	Main Access Decline	6.5	5.5	1,300	–
	Vertical Development				
	Ventilation Raise	–	–	457	3 m diameter
Kamoa Nord	**Lateral Development**				
	Conveyor Decline	6.5	5.5	960	–
	Main Access Decline	6.5	5.5	960	–
	Vertical Development				
	Ventilation Raise	–	–	229	3 m diameter
Total	**Lateral Development**	–	–	16,120	–
	Vertical Development	–	–	3,488	–

The ground support criteria utilized for all development includes threaded rebar and wire mesh with a bolt spacing of 1.5 m. Cable bolts are provided at all intersections where the LHDs load into the feeder-breaker. The ground support design package is at a conceptual level and generalized to meet the minimum support requirements.

Mine-to-Mill Optimization (Blasting)

The drill-and-blast designs for the room-and-pillar stopes in different sections in Kamoa are conceptual designs based on a drill hole spacing suited to the preliminary geotechnical information available. The general design assumes that the overall number of holes drilled and lengths are suitable to achieve the desired fragmentation. The drill and blasting design parameters are presented in Table 16-9. Three-boom jumbos are utilized for all drilling within the stoping panels.

Table 16-9: Drill Blast Design for Primary Drifts in Stoping Panels (5 m high by 5 m wide)

Drill hole Diameter	45.0 mm
Round Length	3.66 m
Number of Holes in Round	60
Explosive Type	ANFO
Powder Factor	1.94 kg/cu.m.

The slashing rounds used to widen the main drifts within the stoping panels consist of 48 parallel holes of 3.66 m length as shown in Table 16-10.

Table 16-10: Drill Blast Design for Slashing in Stoping Panels (5 m high by 5 m wide)

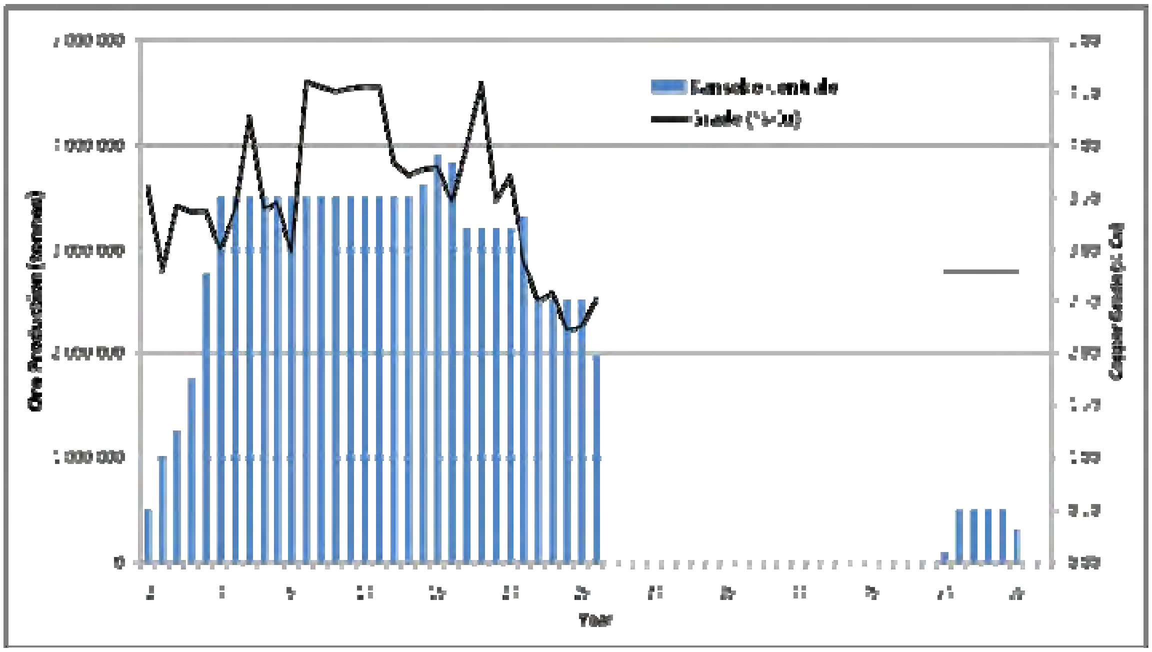
Drill hole Diameter	45.0 mm
Round Length	3.66 m
Number of Holes in Round	48
Explosive Type	ANFO
Powder Factor	1.64 kg/cu.m

16.4.5 Underground Mine Operations

Underground Mine Production Sequence and Schedule

The production sequences for the different sections of the Kamoa Project are based on achieving a combined production rate of 5 Mtpa. The initial target for mining in Kamoa is the Kansoko Centrale Section, which hosts the largest amount of indicated resources with no supergene enrichment. This schedule enables the mining of the best known resource at the start of the project while additional information is gathered about the rest of the areas. The Kansoko Centrale Section is currently scheduled for mining at rates up to 3.9 Mtpa. The production schedule and grades for the Kansoko Centrale Section for the 5 Mtpa case are presented in Figure 16-28.

Figure 16-28:Production Schedule from Kansoko Centrale for 5 Mtpa Case



The layout of the room-and-pillar stoping panels in the different sections provides the flexibility to mine selective blocks and maintain haulage and ventilation routes. The overall plant feed schedule includes the extraction of all mineralized stope, mineralized development, and mineralized tertiary pillar. The life-of-mine tonnes and grade for each section are presented in Table 16-11.

Table 16-11:Underground Production and Grade by Section

Section	Mined Resource Tonnes	Cu%
Kansoko Centrale	98,052,242	3.62
Kansoko Sud	53,399,653	3.08
Kansoko Nord	59,050,113	2.76
Kamoa Sud	23,548,510	3.00
Kamoa Nord	60,837,507	3.10
Total	**294,888,025**	**3.15**

The production schedule is based on mineral resources. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Production rate fluctuations in the different sections can be handled by redistributing the mobile equipment and personnel as required.

Mine Equipment Requirements

The equipment requirements for the each section of the Kamoa Project are split into two categories, fixed equipment and mobile equipment. The equipment requirements for each category are estimated at a conceptual level of accuracy and cover the major components for an overall 5 Mtpa operation, with the equipment being apportioned between the different sections based on the production rates. The following are the design criteria for sizing, selection, and quantifying fixed and mobile equipment.

- Mining Method
- Mine Production Rate of 5 Mtpa
- Ventilation Requirements
- Mine Design Criteria.

A list of major fixed equipment by category for all sections of the Kamoa project is presented in Table 16-12.

Table 16-12: Fixed Mine Equipment

MATERIAL HANDLING	**WATER HANDLING**
Stamler U/G Feeder Breaker BF-38C-69D-79C	*Main Dewatering Pumps*
KC_CV_UPPER_Conveyor	Slurry Pump (incl. valves and fittings)
KV_CV_LOWER_Conveyor	*Clarifying Sump Mud Pumps*
KC_DECLINE_Conveyor	Slurry Pumps (incl. valves and fittings)
CV_PANEL_Conveyor	*Upper Service Water Pump Station*
Metal Detector	Pump and Motor (incl. valves and fittings)
Belt Magnet	*Main Service Water Pump Station*
Belt Scale	Pump and Motor (incl. valves and fittings)
Kansoko_SudA_Decline_CV	Gland Water Pump (incl. valves and fittings)
Kansoko_SudA_Panel_CV	*Development Pumps*
Kansoko_SudB_Decline_CV	Dirty Water Pump
Kansoko_SudB_Panel_CV	Dirty Water Pump
	Face Pumps
COMPRESSOR PLANT	
Air Compressors	**UNDERGROUND SHOP**
Air Pre-filter	Shop Fixed Equipment and Tools
Cooling Water Pump	
Purge Tank, Filter and Water Monitor	**FUEL BAY**
Heat Exchanger & Cooling Fan	Fuel Bay Fixed Equipment and Storage Tanks
Refrigerated Dryer	
Receiver Tank	**SAFETY AND MISCELLANEOUS**
Receiver Auto Drain	Hand-held Drills
	Engineering Equipment (Survey, Lasers, Etc.)
VENTILATION	Maintenance Shop Equipment
Main Intake Fan #1 - 1,000 HP	Shop Monorail Crane - 25T
Main Intake Fan #2 - 1,000 HP	Shop Monorail Crane - 10T
Development/Production Fan - 100 hp	Jib Crane - 5T
Development/Production Fan - 50 hp	Fire Extinguisher
Air Door	First Aid Kit
Shop Air Door - Roll-up	Miscellaneous First Aid Supplies
	Mine Rescue Equipment for 15 Persons
ELECTRICAL	Cap Lamps & Chargers
Main Substation	Self Rescuers
U/G Substation	Miscellaneous Sanitary Supplies and Units
Ring Feed Units	Refuge Chamber
Mine Load Centers	
Main Vent Fan Substation	
Workshop Substation	
Compressor Substation	
Crusher Substation	
Leaky Feeder Radio	
System Data and Control (SCADA)	

The mobile equipment selected for the different sections at Kamoa is based on the criteria above and information from similar projects . The mobile equipment quantities are estimated at a conceptual level and based on historic performance rates and number of crews scheduled throughout the production schedule. The equipment quantity calculation includes projected equipment availability and spares. The overall quantities required in each section fluctuate over the life-of-mine to match the production schedule requirements at any given time.

The equipment rebuild and replacement methodology utilized for this concept study assumes the equipment is rebuilt three years after purchase then replaced three years later. Based on historical rebuild and replacement schedules, a factor of 15% of the total initial equipment purchase cost, applied to each year conservatively, covers the rebuild and replacement costs.

Table 16-13 provides a list of the selected equipment, maximum quantity required life-of-mine, and initial equipment requirements during preproduction. The total quantity required reflects the total amount of equipment for the life-of-mine and does not include replacement equipment.

Table 16-13: Mobile Equipment (Maximum Initial Operating Quantities)

Mobile Equipment (two crew)	Maximum Quantity	Initial Quantity
Toro Axera 8-360 / DD530 Jumbo Drill	6	4
Toro Axera 7-260 / DD420-60 Jumbo Drill (decline development)	3	3
Atlas Copco Boltec MC - Rockbolter - 1 Boom	3	3
LHD Atlas Copco ST-1030	12	4
Getman 324N Rock Scaler	3	3
LHD, 5.4 m^3 Sandvik LH514	3	–
Haul Truck - Sandvik TH540 (backfill)	5	–
Maclean TM-3 Concrete Shotcrete Transit	3	2
MaClean SS-3 Shotcrete Placer Truck	3	2
Explosives Truck / Jumbo, AnfoMaclean Engineering Ac-3 Anfo Charger	3	3
Scissor Lift TruckMaclean Engineering Mine-Mate Sl-3	6	3
Tractor (General purpose) Kubota - Rtv900 - 4x4	6	6
U.G. Road Grader Caterpillar 12 m	3	1
Cassette Carrier Maclean Engineering Cs-3 - Approx 5 Tonne Payload	3	3
Fuel Cassette Maclean Engineering Cs-3 Fuel Cassette - 5,000 Liter Tank	3	3
Fuel/Lube Cassette Maclean Engineering Cs-3 Fuel / Lube Cassette - 2,000 Liter Tank	3	3
Flatbed Cassette Maclean Engineering Cs-3 2.4 Metre Wide Flatbed Cassette	3	2
Water Spraying Cassette	3	3
Backfill Jammer	3	–
Atlas Copco Cabletec LC Cable Bolter	1	–
Subtotal	**78**	**48**

The total ventilation requirements for operating this fleet of equipment can be met with the current accesses to the surface. Ventilation requirements will be higher in the expansion cases because of the larger operating mobile fleets. A detailed ventilation plan should; therefore, be developed to address the ventilation requirements in different sections and necessary intake and exhaust raises should be planned.

16.5 Production Schedule

The open pit schedule for Years -1 to 3 is shown in Table 16-14.

The open pit schedule for Years -1 to 3 is shown in Table 16-14. The processing schedule for Years -3 to 61 are shown in Table 16-15. The production schedule is based on mineral resources. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 16-14:Open Pit Production Schedule

		Total/Year	**-1**	**1**	**2**	**3**
Total Mining	kt	48,742	15,000	14,812	15,000	3,930
Waste Mining	kt	44,239	14,812	13,000	13,500	2,927
Strip Ratio	W:O	9.8	78.6	7.2	9.0	2.9
Plant Feed	kt	4,503	188	1,812	1,500	1,003
	% Cu	2.96	2.53	2.58	3.01	3.66

Table 16-15: Processing Production Schedule

	Year No	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026
	Total/Year	**-3**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**
Processing kt	299,388	–	–	–	4,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000
Cu%	3.14	–	–	–	3.02	3.20	3.48	3.22	3.46	4.12	3.48	3.51	3.19	4.26
Cu Rec %	84.8	–	–	–	81.6	83.7	84.6	85.1	85.8	87.4	85.9	86.0	85.0	87.6
Concentrate kt	24,384	–	–	–	312	416	458	418	453	549	455	460	413	570
Concentrate Cu%	32.7	–	–	–	31.6	32.2	32.2	32.8	32.8	32.8	32.8	32.8	32.8	32.8
Concentrate Cu kt	7,984	–	–	–	99	134	147	137	148	180	149	151	136	187
Concentrate Cu Mlb	17,601	–	–	–	217	295	325	302	327	397	329	332	299	412
Smelter Recovery %	98	–	–	–	98	98	98	98	98	98	98	98	98	98
Payable Cu kt	7,801	–	–	–	96	131	144	134	145	176	146	147	132	182
Payable Cu Mlb	17,198	–	–	–	212	288	318	295	320	388	321	325	292	402
Acid Production kt	16,190	–	–	–	207	276	304	277	300	365	302	305	274	378
	Year No	**2027**	**2028**	**2029**	**2030**	**2031**	**2032**	**2033**	**2034**	**2035**	**2036**	**2037**	**2038**	**2039**
	Total/Year	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**	**20**	**21**	**22**	**23**
Processing kt	299,388	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000
Cu%	3.14	4.28	4.27	4.26	4.31	4.34	3.80	3.66	3.70	3.71	3.32	3.93	4.10	3.45
Cu Rec %	84.8	87.7	87.7	87.6	87.7	87.8	86.7	86.3	86.4	86.5	85.4	87.0	87.3	85.8
Concentrate kt	24,384	572	571	569	577	581	503	482	488	489	433	522	546	452
Concentrate Cu%	32.7	32.8	32.8	32.8	32.8	32.8	32.8	32.8	32.8	32.8	32.8	32.8	32.8	32.8
Concentrate Cu kt	7,984	188	187	187	189	190	165	158	160	160	142	171	179	148
Concentrate Cu Mlb	17,601	413	413	411	417	420	364	348	352	354	313	377	394	326
Smelter Recovery %	98	98	98	98	98	98	98	98	98	98	98	98	98	98
Payable Cu kt	7,801	183	183	182	185	186	161	154	156	157	139	167	175	145
Payable Cu Mlb	17,198	404	403	402	407	410	355	340	344	345	306	369	385	319
Acid Production kt	16,190	380	379	378	383	385	334	320	324	325	288	347	362	300

	Year No	**2040**	**2041**	**2042**	**2043**	**2044**	**2045**	**2046**	**2047**	**2048**	**2049**	**2050**	**2051**	**2052**
	Total/Year	**24**	**25**	**26**	**27**	**28**	**29**	**30**	**31**	**32**	**33**	**34**	**35**	**36**
Processing kt	299,388	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000
Cu%	3.14	3.58	3.06	2.58	2.53	2.22	2.57	2.90	3.15	3.66	2.80	2.58	2.56	2.48
Cu Rec %	84.8	86.1	84.6	82.7	82.4	80.7	82.6	84.0	84.9	86.4	83.6	82.7	82.6	82.1
Concentrate kt	24,384	470	395	326	318	273	323	372	409	483	358	325	323	310
Concentrate Cu%	32.7	32.8	32.8	32.8	32.8	32.8	32.8	32.8	32.8	32.8	32.8	32.8	32.8	32.8
Concentrate Cu kt	7,984	154	130	107	104	89	106	122	134	158	117	107	106	102
Concentrate Cu Mlb	17,601	340	286	235	230	197	234	269	295	349	258	235	233	224
Smelter Recovery %	98	98	98	98	98	98	98	98	98	98	98	98	98	98
Payable Cu kt	7,801	150	127	104	102	87	103	119	131	155	115	104	103	99
Payable Cu Mlb	17,198	332	279	230	225	193	228	262	289	341	253	229	228	219
Acid Production kt	16,190	312	263	216	211	181	215	247	271	320	237	216	214	206
	Year No	**2053**	**2054**	**2055**	**2056**	**2057**	**2058**	**2059**	**2060**	**2061**	**2062**	**2063**	**2064**	**2065**
	Total/Year	**37**	**38**	**39**	**40**	**41**	**42**	**43**	**44**	**45**	**46**	**47**	**48**	**49**
Processing kt	299,388	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000
Cu%	3.14	2.35	2.57	2.67	2.41	2.35	2.59	2.59	2.60	2.74	2.61	2.79	2.83	3.11
Cu Rec %	84.8	81.5	82.6	83.1	81.8	81.5	82.7	82.7	82.7	83.4	82.8	83.6	83.7	84.8
Concentrate kt	24,384	292	323	339	301	292	326	327	328	348	330	355	361	402
Concentrate Cu%	32.7	32.8	32.8	32.8	32.8	32.8	32.8	32.8	32.8	32.8	32.8	32.8	32.8	32.8
Concentrate Cu kt	7,984	96	106	111	99	96	107	107	107	114	108	117	118	132
Concentrate Cu Mlb	17,601	211	234	245	217	211	236	237	237	251	238	257	261	291
Smelter Recovery %	98	98	98	98	98	98	98	98	98	98	98	98	98	98
Payable Cu kt	7,801	93	104	109	96	93	105	105	105	111	106	114	116	129
Payable Cu Mlb	17,198	206	228	239	212	206	230	231	231	246	233	251	255	284
Acid Production kt	16,190	194	215	225	200	194	217	217	218	231	219	236	240	267

	Year No	2066	2067	2068	2069	2070	2071	2072	2073	2074	2075	2076	2077
	Total/Year	**50**	**51**	**52**	**53**	**54**	**55**	**56**	**57**	**58**	**59**	**60**	**61**
Processing kt	299,388	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	5,000	388
Cu%	3.14	2.97	2.91	2.91	2.97	2.86	2.73	2.73	2.73	2.73	3.02	3.10	3.10
Cu Rec %	84.8	84.3	84.1	84.1	84.3	83.9	83.3	83.4	83.4	83.4	84.5	84.7	84.7
Concentrate kt	24,384	382	373	373	381	365	347	348	348	348	390	400	31
Concentrate Cu%	32.7	32.8	32.8	32.8	32.8	32.8	32.8	32.8	32.8	32.8	32.8	32.8	32.8
Concentrate Cu kt	7,984	125	122	122	125	120	114	114	114	114	128	131	10
Concentrate Cu Mlb	17,601	276	270	270	276	264	251	251	251	251	282	289	22
Smelter Recovery %	98	98	98	98	98	98	98	98	98	98	98	98	98
Payable Cu kt	7,801	122	120	120	122	117	111	111	111	111	125	128	10
Payable Cu Mlb	17,198	270	264	264	269	258	245	245	245	245	275	283	22
Acid Production kt	16,190	254	248	248	253	243	230	231	231	231	259	266	21

17.0 RECOVERY METHODS

17.1 Introduction

This section on recovery methods incorporates assumptions, analysis and findings of the 2012 PEA.

The information relevant to the plant design supporting the financial analysis prepared during the 2012 PEA is included in this section and has not been updated because the QPs consider that the 2012 PEA is still current, given the assumptions supporting the outcomes remain reasonable. The effective date of the 2012 PEA results remains 5 September 2012.

The concentrator is a 5 million metric tonne per annum operation with an MF2 circuit configuration. The plant copper feed grade (LOM average) is 3.30 % and is designed to achieve 85.4% recovery at 32.8% Cu concentrate grade (mass pull approximately 8.6%).

A three-stage crushing circuit feeds the primary mill feed silo. The primary and secondary ball mills operate in closed circuit with hydrocyclones. The flotation circuit consists of roughers and scavengers with a re-grind mill between the rougher and scavenger stages. The cleaner circuit consists of cleaners, scavenger cleaners and re-cleaners. The cleaner circuit incorporates two concentrate regrind stages. The final concentrate is thickened before it is pumped to the concentrate filter. The concentrate shed provides 24 hour storage for the filtered concentrate before it is fed to the smelter. The scavenger tails and multiple non-float streams from the scavenger cleaner circuit report to final tails.

The smelting process is based on the use of direct to blister flash smelting technology (DBF). For slag cleaning, a two-stage electric furnace process is applied. The smelter has a concentrate smelting capacity of 660,000 tpa, which corresponds to a copper product capacity of 182,260 tpa.

In the DBF concept, copper concentrate is processed by flash smelting to produce blister copper (98% Cu) in one stage. Blister copper is transferred via launders to anode furnace treatment.

Slag obtained from the flash smelting furnace is treated in an electric slag cleaning furnace (SCF) by coke reduction to reduce the copper content down to 4%. Metallic copper thus formed is laundered to an anode furnace.

Final slag cleaning and cobalt recovery take place in a second electric furnace, the cobalt reduction furnace (CoRF), in series. The slag is treated by coke reduction to lower the copper and cobalt contents to 0.6% and 0.1%, respectively. Some concentrate mixture is injected to the bath to increase the sulphur content of the Cu-Fe-Co alloy settling to the bottom of the furnace. The Cu-Fe-Co alloy and waste slag are granulated directly from the furnace using a hot quench granulation method. The granulated alloy is saleable to a cobalt refinery.

Cobalt recovery is referred in this section. It is included since it is known that cobalt can, and has, occurred in Katanga and Copperbelt ores. However, cobalt has not yet been identified in the current test phase. The potential for economic recovery of cobalt will be addressed in the next phase.

17.2 Design Criteria

17.2.1 RoM Composition

The RoM compositions listed are averages of samples from the Kansoko Nord, Kansoko Centrale and Kamoa South areas. The design head grade, as shown in Table 17-1, was based on the mine schedule.

Table 17-1: Feed Composition

Average Feed Head Grade	Unit	Value (Design)
Kamoa South		
Cu	%	3.30
As Cu	%	0.0
Total S	%	2.16
Al	%	4.5
Si	%	18.6
P	%	0.5
S	%	0.8
Cl	%	0.0
K	%	2.8
Ca	%	0.2
Ti	%	0.5
V	%	0.0
Cr	%	0.0
Mn	%	0.0
Fe	%	5.5
Head Grade Design (Cu)	%	3.30

Note: No cobalt is reported in this table as there are currently no Mineral Resource estimates for the element and cobalt deportment has not been tracked in flotation testing. However, Co is included in the PEA process design in case of future recovery potential. Cobalt can, and has, occurred in Katanga and Copperbelt mineralization as a recoverable element.

17.2.2 Availability, Utilization, and Throughput

The key availability criteria used to size the process equipment is listed in Table 17-2. Availability is defined as time relative to the total number of hours in a year.

Table 17-2: Design Criteria

Average Feed Head Grade	Unit	Value (Design)
Overall Crusher Availability	%	85
Crusher Operating Days	d/wk	7
Overall Mill Availability	%	91
Smelter Availability	%	89

These availability figures are in line with industry norms for these types of operations.

17.3 Concentrator Description

The concentrator is described as if it exists (present tense) only for clarity of the explanation. No concentrator currently exists at the Kamoa project site.

17.3.1 Receiving, Crushing, and Mill Feed Silos

RoM mineralized material is conveyed from the primary jaw crusher, situated underground, to a surface RoM bin (500 t). Mineralized material is withdrawn from the bin via a vibrating grizzly feeder. The grizzly oversize is fed to a jaw crusher. Grizzly undersize and jaw crusher product are combined and conveyed to the primary screen. The primary screen oversize (+20 mm) is fed to the secondary cone crusher. The secondary and tertiary cone crusher products, together with primary screen undersize, are transferred to the secondary screen. The secondary screen oversize feeds the tertiary cone crusher. To ensure choke feed conditions, each crusher is preceded by a surge bin and is fed with a variable speed belt feeder.

The secondary screen undersize (-12 mm) feeds the mill feed silos (2 silos with a total capacity of 24 hours). Mineralized material is withdrawn from the silo to feed the primary ball mill. There is a primary mill feed belt sampler and weightometer on the feed conveyor to the mill.

All fine spillage from the crushing circuit is pumped to the primary mill discharge sump whilst coarse particles are prevented from entering the spillage pump and can be cleaned up manually.

17.3.2 Primary Milling

Crushed mineralized material (-12 mm) is fed into a ball mill (overflow mill, 25 ft Ø x 38 ft EGL, 12.5 MW installed power). The motor has a fixed speed twin pinion drive with an operating speed of 75% net critical speed (NCs). The design ball charge is 30% with a maximum operating charge of 35%.

The mill discharge trommel screen oversize reports to the scats bin and the primary mill product (trommel undersize) gravitates to the mill discharge sump. The trommel undersize is diluted with process water before being pumped to the classification cyclone cluster with a variable speed cyclone feed pump. The mill operates in closed circuit with hydrocyclones with a 250% nominal recirculating load (design 300%). The cyclone overflow (P_{80} of 80 µm) gravitates to the primary rougher feed surge tank at a slurry density of about 30% solids, equivalent to 1.3 tonnes per cubic metre. Dispersant is added to the primary ball mill feed.

17.3.3 Secondary Milling

Primary rougher flotation tails are pumped to the secondary ball mill discharge sump. The ball mill (overflow mill, 19 ft diameter x 29 ft EGL, with a single pinion fixed speed

5.1 MW drive) motor operating speed is 75% NCs and the design ball charge is 30% with a maximum operating charge of 35%.

The mill discharge trommel screen oversize reports to the scats bin and the secondary mill product (trommel undersize) gravitates to the mill discharge sump. The trommel undersize is diluted with process water before being pumped to the secondary classification cyclone cluster. The mill operates in closed circuit with hydrocyclones with a 250% nominal recirculating load (design 300%). The cyclone overflow (P80 of 45 µm) gravitates to the scavenger feed surge tank at a slurry density of 30% solids, which is equivalent to 1.3 tonnes per cubic metre.

17.3.4 Primary and Secondary Milling

The media used for both mills will be high chrome steel but the secondary mill balls will be smaller diameter than the primary mill balls. Steel balls are added to both mills using a dedicated ball kibble, magnet, and hoist arrangement. This will be a semi-automated operation and steel will be added as required. Spillage from the respective milling circuits will report to the corresponding mill discharge sumps. Scats from both mills, typically worn media and a small amount of oversize rock particles, are discharged into respective kibbles for circuit removal.

17.3.5 Flotation Circuit General

Froth collection launders and piping will be designed to incorporate flexibility with regards to the routing of the various streams.

Flotation tailings sumps are sized for 90 second residence times. Spillage generated within the respective rougher flotation circuits will be pumped to the corresponding rougher feed surge tanks or the mill discharge sumps. Spillage from the cleaner circuit will be pumped to the regrind mill feed sump. Flows will be measured using magnetic flow meters where applicable. All flotation cells will incorporate a master/slave configuration for dart valves. The air flow into each cell will be controlled via the SCADA system.

17.3.6 Rougher and Scavenger Flotation Circuit

Primary mill cyclone overflow reports to the rougher feed surge tank with a residence time of 15 minutes at 70% capacity. This is sufficient residence time to allow for reagent conditioning and to smooth the flotation feed flow rate in the short term i.e. mitigation of volumetric surges. Collector is added to the cyclone overflow launder ahead of the surge tank and frother is added to the surge tank discharge into the first rougher cell.

The flotation feed is pumped via two pumps (1 operating, 1 standby – both VSD) to rougher cell 1. A vezin sampler arrangement is configured to sample the flotation feed. Secondary mill cyclone overflow reports to the scavenger feed surge tank. Frother and collector are also added to this tank. The flotation feed is pumped via two pumps (one operating, one standby – both variable speed drives) to the first scavenger cell.

The primary rougher bank consists of seven, 150 m^3 cells with a collective residence time of 50 minutes (mass pull approximately 20%). The primary rougher tails are pumped to the secondary mill discharge sump. The scavenger bank consists of five, 200 m^3 cells with a collective residence time of 40 minutes (mass pull approximately 7%). The scavenger tails are pumped to the final tails thickener.

17.3.7 Concentrate Regrind Mills

Rougher Concentrate

The rougher concentrate reports to the regrind mill circuit. The type of mill selected for this application is the M10000 horizontal IsaMill with a fixed speed 3000 kW drive. The regrind mill duty and equipment selection will be confirmed at a later study phase after detailed testwork has been performed.

The combined rougher concentrates are pumped via froth pumps to a tank before being pumped to densifying hydrocyclones. The densifying cyclones produce an underflow density of 50% solids, suitable for IsaMill feed. The cyclone overflow and the regrind mill product are combined in a sump before being pumped to the cleaners. The media used will be ceramic media (MT1). The approximate consumption is 10 g/kWh; however, this will be confirmed with testwork. Media is supplied in 2 t bags and media management is incorporated into the M10000 IsaMill facility design. The P_{80} of the rougher regrind mill circuit is 15 µm.

Scavenger Concentrate

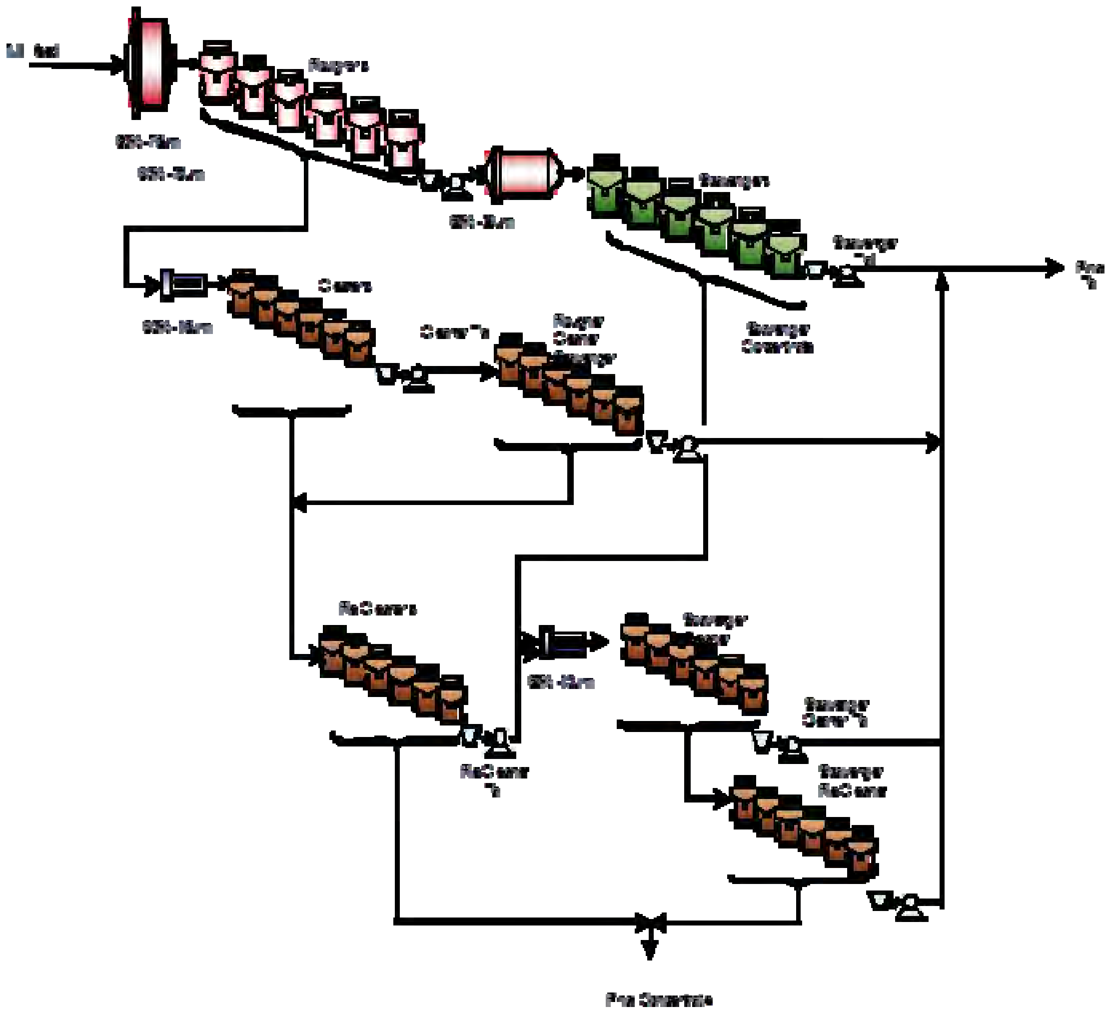
The scavenger concentrate and the re-cleaner tail report to the second regrind mill circuit. The type of mill selected for this application is expected to be comparable to that selected for the rougher concentrate regrind. The regrind mill duty and equipment selection will be confirmed at a later study phase after detailed testwork has been performed.

The combined scavenger concentrates are pumped via froth pumps to a tank before being pumped to densifying hydrocyclones. The densifying cyclones produce an underflow density of 50% solids, suitable for IsaMill feed. The cyclone overflow and the regrind mill product are combined in a sump before being pumped to the cleaners. The media used will be ceramic media (MT1). The approximate consumption is 10 g/kWh; however, this will be confirmed with testwork. Media is supplied in 2 t bags and media management is incorporated into the M10000 IsaMill facility design. The P_{80} of the scavenger regrind mill circuit is 10 µm.

Cleaner Flotation Circuit

The rougher and scavenger concentrates report separately to cleaning circuits (cleaning and scavenging with re-cleaning at both stages), as shown in Figure 17-1. Frother and collector are added to the head of each bank. The rougher regrind mill product and the densifying cyclone overflow are combined and pumped to the rougher cleaner bank, consisting of a primary bank and a scavenger bank. The rougher scavenger cleaner tails are pumped to the final tails thickener. The primary and scavenger concentrates are combined and pumped to the re-cleaner bank. The re-cleaner concentrate is pumped to the final concentrate thickener. The re-cleaner tails, along with the scavenger concentrate, are transferred to the scavenger regrind mill, the product of which is fed to the scavenger cleaner bank. The scavenger cleaner concentrate is re-cleaned and is pumped to the final concentrate thickener. The tails from both scavenger cleaner banks (cleaner and re-cleaner) are pumped to the final tails thickener.

Figure 17-1: Flotation Circuit



Note: Figure courtesy Ivanplats, sourced from XPS, 2011

17.3.8 Thickening Circuits

Concentrate Thickener and Filter

The final concentrates are pumped to the concentrate thickener via a trash screen and metal accounting samplers. The concentrate thickener is a high rate, 21 metre diameter thickener. Flocculant at 11 g/t is added to the thickener feed. The overflow from the thickener gravitates to the process water tank. Two peristaltic pumps (one operating, one standby – variable speed) pump the thickener underflow to the filter feed tank at 65% solids. The underflow slurry can also be recycled to the thickener during start-up or when the filter feed tank cannot receive feed. The filter feed tank (20 hours residence time) allows for storage of the concentrate before batch feeding to the concentrate filter. Spillage from the thickener area is pumped to the thickener feed well.

Tailings Thickener

The scavenger tails and the various scavenger tails from the cleaner circuits are pumped to the tailings thickener via a trash screen and metal accounting samplers. Various reagent spillage streams are also pumped to the tailings thickener. The tails thickener is a high rate, 53 metre diameter thickener. Flocculant at 30 g/t is added to the thickener feed. The overflow from the thickener gravitates to the process water tank. Two stage centrifugal pumps deliver the thickener underflow to the tailings dam at 55% solids. The slurry can also be recycled to the thickener during start-up or when the tailings system is unavailable. Spillage from the thickener area is pumped to the thickener feed well. The final tails stream is sampled for metal accounting purposes before it enters the tailings thickener.

17.3.9 Services and Utilities

Provision has been made for the mixing and supply of the necessary reagents for flotation. Individual peristaltic pumps and piping are installed for each addition point of each reagent.

Raw water from local boreholes or surface sources (still to be determined) is collected and stored in a raw water dam. Filtration and treatment plants produce a range of water qualities as required for potable water, gland seal water, boiler feed water, cooling water, granulation water and process water usage. Distribution systems for each water type are included, ensuring delivery of sufficient quantity at the required pressure.

Dedicated blowers supply manifold air for the flotation cells.

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 Smelter

The Kamoa smelter technology will be direct to blister flash smelting (DBF). For slag cleaning a two-stage electric furnace process will be used. The smelter will have a maximum concentrate capacity of 660 kt/a with a metal content of 186 kt Cu/a.

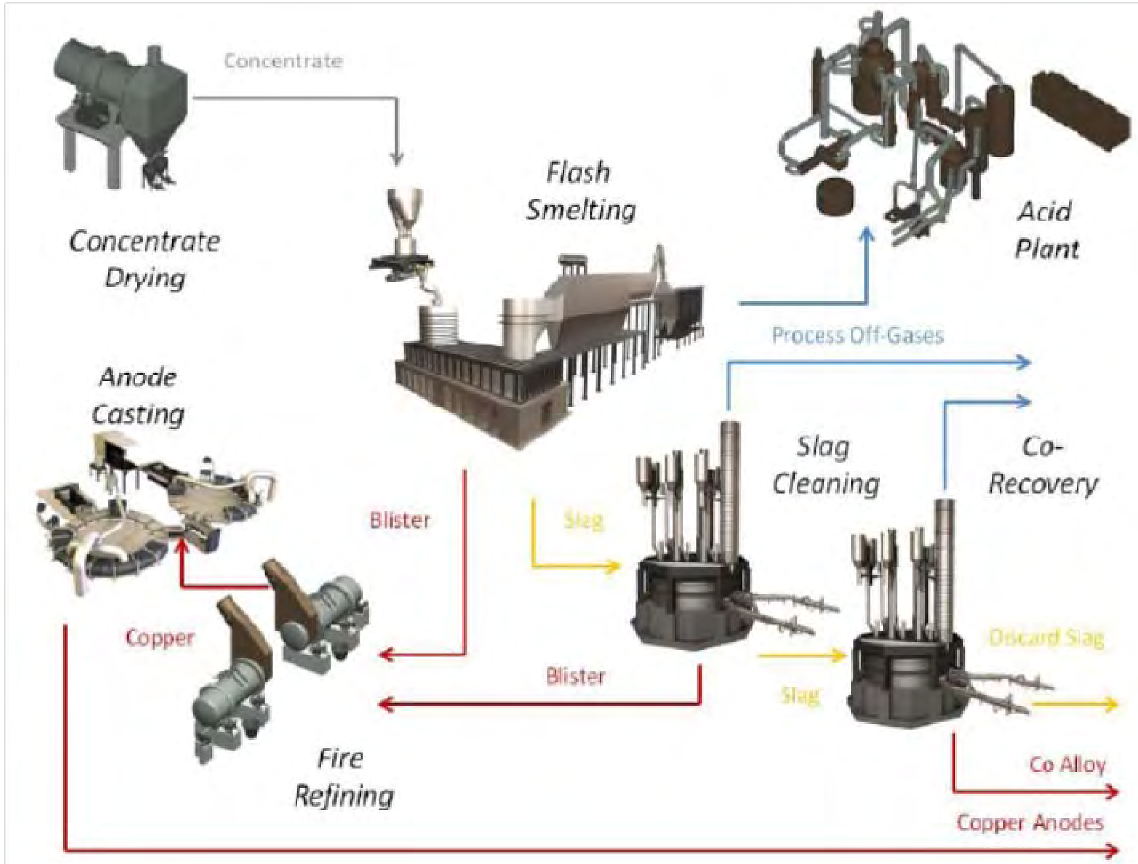
In the DBF concept, bone-dry copper concentrate will be processed by flash smelting to produce blister copper in one stage. Blister copper will be transferred via launders to anode furnace treatment.

Slag obtained from the flash smelting furnace will be treated in an electric slag cleaning furnace (SCF) by coke reduction to lower the copper content down to 4%. Metallic copper thus formed will be laundered to an anode furnace.

After refining in the anode furnace the copper will be cast as anodes ready for sale.

Provision for cobalt recovery is made in the process flow-sheet should metallurgical tests indicate potential for economic recovery of cobalt. AMEC notes that the cobalt grade and metallurgical recoveries would have to be estimated before cobalt can be included in any production plans or economic analysis. Final slag cleaning and cobalt recovery will take place in an electric CoRF. The CoRF may not be an economical option due to the low cobalt content of the concentrates and this concept will be revisited at the next study phase. Conceptually, the slag will be treated by coke reduction to lower the copper and cobalt contents to 0.6% and 0.1%, respectively. Some concentrate mixture will be injected to the bath to increase the sulphur content of the Cu-Fe-Co alloy settling to the bottom of the furnace. The Cu-Fe-Co alloy and waste slag will be granulated directly from the furnace. The conceptual smelter flow sheet is depicted in Figure 17-2.

Figure 17-2: Flow Sheet of the Direct Blister Process



Note: Figure courtesy Outotec, 2011

17.4.1 Concentrate Drying

The wet concentrate mixture (moisture 12%) and limestone flux (moisture 4%) are conveyed from their respective storage areas to the dryer area and fed by a screw feeder to a rotary steam dryer. Drying of wet feed material utilizes indirect heating with steam as the energy source. The dryer is rotated with an electrical motor with a frequency converter.

Heat for drying is supplied by saturated waste heat boiler steam at a pressure of 20 bar. The dryer exhaust gas handling system consists of a bag filter, exhaust gas fan, ducting and an exhaust stack. The dryer product and dust from the bag filter are transferred with a drag conveyor to a vertical pneumatic conveyor system and further to a feed mixture bin ahead of the smelter. Part of the feed mixture is transported by a pneumatic conveyor to the Cobalt Recovery Area to be used as a sulphidising agent in the cobalt recovery furnace.

17.4.2 Flash Smelting

Direct Blister Furnace

The flash smelting furnace consists of a vertical reaction shaft, a horizontal furnace settler and a vertical uptake shaft. The refractory lining of the furnace sections is water cooled.

The furnace feed material is discharged at a controlled rate from the feed mixture bin onto an air slide by a loss-in-weight feeder (LIW). A similar feeding and control arrangement is used for the flue dust collected from the DBF gas cooling and cleaning equipment.

The air slide conveyor transports the mixture of dried concentrate, recycled flue dust and limestone flux to the concentrate burner.

Special cooling elements are installed in the reaction shaft walls, settler walls, settler roof, uptake shaft and tapping hole area. Cooling water flows in a closed circuit and is continuously cooled by heat exchangers.

The bottom of the settler is cooled with hearth cooling air fans to prevent the blister penetrating in the brick lining. The settler part of the furnace is provided with oil burners at wall and roof to control the furnace heat balance make up and for heat-up/holding purposes. The burners are operated with air.

Flash Smelting Process

The dry charge mixture and the oxygen enriched process air form an even suspension in the reaction shaft. As the suspension flows downwards in the reaction shaft the concentrate particles dissociate, ignite and undergo controlled partial oxidation generating a large amount of heat which results in melting of the material to fine droplets.

The reaction shaft temperature is controlled by the oxygen enrichment of process air and by introducing some fuel oil through the reaction shaft lance. To compensate the heat losses of the settler, fuel oil is fed through the oil burners located in the roof of the settler. Oxidation degree (i.e. the sulphur content of blister copper) is controlled by the total oxygen/feed mixture ratio.

At the bottom of the reaction shaft, the direction of the gas-particle stream is changed by 90° and the gas flows through the horizontal settler. Most of the particles are settled out of the gas stream and form a molten bath where the oxidation reactions are completed. As a result the immiscible slag and blister phases separate and form two layers in the settler according to their specific densities. A separate settling unit is not needed.

The process is operated at highly oxidizing conditions resulting in blister copper with a low sulphur content. Consequently, the copper content of slag is quite high. However, more than half of the primary copper in the concentrate is recovered in the DBF blister product.

The blister tapping holes are located at the sidewall and end wall of the settler. Blister is periodically tapped through launders directly to anode furnaces. The slag is tapped through the tapping holes in the side wall of the settler. The slag is laundered directly into an electric slag cleaning furnace (SCF). The blister launders are of steel plate construction with refractory lining. The slag launders are water cooled copper block launders.

17.4.3 Gas Handling

The process gas flows upward through the uptake shaft and then to a waste heat boiler (WHB) for cooling. Part of the flue dust is separated from the gas stream in the WHB. The gas flows onward to an electrostatic precipitator (ESP) where the remaining fine dust is recovered. The dust-free gas is piped to a sulphuric acid plant (SAP).

During cooling, the oxidic flue dust tends to form sulfates. To control the rate of sulphate production, sulphation air and oxygen are added in the process streams.

DBF Ventilation Gas Handling

The slag and the blister launders and all the tapping hole openings are hooded for collection of fume and dust containing gases. The gases are collected and combined with the anode furnace off-gases from the reduction and casting stages. The off-gases are drawn by a fan for de-dusting in the bag filter and are then directed to atmosphere.

The waste heat boiler and the electrostatic precipitator dust hopper bottoms are provided with drag conveyors for collecting and conveying the flue dust. All the flue dust formed in the process is circulated back into the furnace.

17.4.4 Slag Cleaning and Cobalt Recovery

Molten slag is transferred from the DBF to the SCF where it is mixed with other solid feed components. The SCF generates blister grade copper for anode production. The molen slag from the SCF then transfers to the CoRF where, again, solid feed components are added. The CoRF generates a granulated Co rich alloy for separate sale and the CoRF slag is the final smelting solid waste.

Furnace (SCF and CoRF) Feed System

The furnaces' feed system consists of five feed bins for coke and solid materials. The bins are filled intermittently by a common feeder system. Material from the feed bin is discharged, weighed and dosed batch-wise into the furnace using vibrating feeders. The weighed coke batch is dropped into the furnace via the feed funnels located at the top of the furnace roof. The ventilation air from the furnace roof hoods is collected and blown as afterburning/dilution air into the water-cooled off-gas incinerators.

Slag Cleaning Furnace and Cobalt Recovery Furnace

The electric furnaces are similar in construction consisting of cylindrical brick-lined hearth, charging equipment, water cooling system for the refractory lining, air cooling for the furnace bottom, power supply equipment, ventilation system, gas cooling and dust handling equipment. The furnaces are also provided with tapping holes and launders for blister/alloy and slag.

Electric power is used for compensation of the heat needed for the endothermic reduction reactions, furnace heat losses and melting of the solid material fed into the furnace.

In the first furnace (SCF) the copper content of slag is reduced to 4-5%. To control the copper content slag is periodically tapped from the SCF. Coke is used to reduce ferric iron to ferrous form. Copper separates from the slag phase as metallic droplets and settles to the bottom of the furnace and forms metallic product, which is taken to anode furnaces. The slag is laundered to another electric furnace (CoRF) for final recovery of copper and cobalt.

In the CoRF the copper and cobalt contents of SCF slag are lowered to waste slag level by coke additions. In order to achieve sufficiently low contents of valuable metals in waste slag some ferrous iron has to be reduced to metallic form. The product obtained from this process step is a Cu-Fe-Co alloy containing some sulphur. The sulphur content of the alloy is controlled by injecting concentrate into the CoRF bath.

Sulphur addition is necessary because it helps to maintain the liquidus temperature of the alloy low enough to ensure trouble free tapping of the alloy from the furnace. The sulphur content also improves the grinding characteristics of the granulated alloy making the granules more brittle.

Feed mixture for sulphidisation is taken from the storage bin by screw feeders to a pressure feeder, from which the material is conveyed with compressed air via tubes to lances immersed into the slag layer. The lances are provided with equipment for moving the lances up and down. Sulphidising is carried out batch-wise, one batch for each slag reduction cycle.

The Cu-Fe-Co alloy is tapped periodically from the CoRF and laundered to granulation. Molten alloy flows through the launder and strikes a refractory spray head, splits up and forms droplets, which fall into a water tank underneath. After quenching, the granules are discharged from the tank by a slurry pump conveying the material to a dewatering system. There is a potential to sell the alloy to a refinery on the adjacent Copperbelt: this has not been fully explored and is at a conceptual stage.

The highly-reduced slag is tapped from the CoRF through the slag tapping holes then via launders into granulation. The granulation is carried out by water jets generated by nozzles located underneath the slag launder. The granulated material settles to the bottom of the granulation pit from where it is lifted by elevators and transported for dewatering treatment.

Off-Gas and Ventilation Gas Handling

The SCF off-gas is first post-combusted and diluted in the off-gas incinerator. The afterburning and dilution of the off-gas is carried out by blowing air and the collected ventilation gases from the furnace roof into the incinerator. The after-burned and diluted SCF off-gas is led further for additional cooling in an evaporative cooler. After that the gas is taken for dust removal in a bag filter and finally discharged via a stack.

The CoRF off-gas is treated like the SCF off-gas, but because of the lower off-gas volume there is no evaporative cooler in the line but the gas is taken directly to a bag filter.

The slag and the blister launders and all the tapping hole openings are hooded for collection of the ventilation gases. All the SCF area ventilation gases are collected and led via the common ventilation gas ductwork to the SCF off-gas handling bag house. Similarly, all the CoRF area ventilation gases are collected and led via the common ventilation gas ductwork to the CoRF off-gas handling bag house.

17.4.5 Anode Furnace Recovery and Casting

Fire Refining

Blister copper tapped from the DBF and SCF contains minor amounts of elements, such as sulphur, oxygen, iron, arsenic.

The purpose of the anode furnace is to refine blister copper and to produce anodes which have the required physical and chemical properties for a subsequent electrolytic refining stage.

The sulphur and oxygen contents o f blister copper are dependent on the operating mode of the previous process step. The more sulphur there is left in the blister, the more oxidation is required in the anode furnace. In this case the DBF process is operated in the range where the sulphur content of blister is 0.3%. The SCF copper is practically sulphur-free.

The refining process proceeds through the following stages: charging, oxidation, slagging, deoxidation (reduction) and casting.

Oxidation is carried out by blowing air through tuyères (gas injection ports) into molten copper. By oxidation it is possible to transfer impurities like sulphur to the gas phase and iron to the slag phase. After oxidation the oxygen content of copper is about 0.8%. The formed slag is skimmed off to prevent the impurities from dissolving back into the molten copper during the reduction stage. The anode furnace slag is fed to the SCF. Excess oxygen is removed by reduction with an oil and steam mixture. When oil and steam is blown through the tuyères into molten copper, it generates carbon monoxide and hydrogen, which act as reducing agents. An optimum content of oxygen in the copper metal phase for subsequent process steps is normally in the range 0.1 to 0.2%.

Anode Casting

Anode casting equipment consists of a casting system, automatic weighing and casting machines, an anode casting wheel with anode moulds and take-off devices and cooling tank units. When the anode furnace is tilted the molten copper flows along the launder into the intermediate ladle which feeds the weighing casting ladle. The casting ladle casts the copper into two casting moulds concentrically located in the casting wheel. When the anodes have been cast the casting wheel turns forwards. The anodes in the moulds turn to the cooling section where water is sprayed onto them. At the take-off section the take-off device lifts the solidified anodes from casting moulds to the cooling tank. Finally the moulds are redressed by a water solution of barium sulphate and they turn under the casting ladles for the next casting stage. From the cooling tank, finished anodes are lifted by a crane in bundles for transportation.

17.4.6 Sulphuric Acid Plant

The function of the sulphuric acid plant is to receive the sulphur dioxide containing process gases from the direct blister and anode furnaces and to produce concentrated sulphuric acid from these gases.

The sulphuric acid process consists of the following four principal steps:

- Gas cleaning of the sulphur dioxide gas.
- Drying of the sulphur dioxide (SO_2) gas.
- Conversion of the sulphur dioxide (SO_2) gas to sulphur trioxide (SO_3) gas according to the chemical reaction: $SO_2 + 1/2\ O_2 <=> SO_3$.
- Absorption of the sulphur trioxide (SO_3) gas by combining with water (H_2O) to form a solution of 98.5% sulphuric acid (H_2SO_4) according to the chemical reaction: $SO_3 + H_2O => H_2SO_4$.

In this study the basic configuration of the sulphuric acid plant is the 3+1 double absorption process. The sulphuric acid plant can be divided into two main sections: gas cleaning and gas contact sections. DBF process gas and anode furnace off-gases (in oxidation and holding stages) are combined into a common gas duct before the sulphuric acid plant. The average gas flow of the incoming process gas to the gas cleaning section varies in the range 47,000–70,000 Nm^3/h (gas flow) with 20–27 volume % sulphur dioxide SO_2, depending on the operation stage of the anode furnace and the sulphur content of the concentrate.

The gas contact section operates with an average flow of 75,000 Nm^3/h at nominally 11 volume % of SO_2, again depending on the sulphur content of the concentrate.

The anode furnace off-gas also serves as an additional oxygen source for the sulphuric acid plant providing the required oxygen for the dioxide to trioxide conversion reaction. During reduction and casting stages atmospheric air is suctioned via the drying tower for this purpose.

The estimated average sulphuric acid production is 727 t/d H_2SO_4. The product quality is 98.5% sulphuric acid.

17.4.7 Services and Utilities

The oxygen requirements of the smelter are met by an “over-the-fence” supply from a vendor cryogenic oxygen plant. The plant is owned and operated by the vendor. A cryogenic plant is necessary to meet the 95% O_2 purity requirements specified by the smelter plant vendor. The small nitrogen requirements for the smelter are also met from this supply.

Fuel oil (LFO) has been assumed as available locally in Lubumbashi and is tankered in to the plant. Plant storage is at one month’s usage.

The closest coke supplier is Hwange, Zimbabwe. On-site storage has been set at one month and utilises a “day bin” of one week capacity.

Limestone is assumed to be available from Ndola Lime, Zambia. On-site storage has been set at one month and utilises a “day bin” of one week capacity.

17.4.8 Equipment Specifications and List

Table 17-3 is the projected equipment requirement overview for the proposed concentrator. This list forms the basis of the concentrator capital cost estimate but it no longer fully reflects the concentrator flowsheet that has emerged (since the time the capital estimation basis was frozen) from the metallurgical testwork program.

Table 17-3: Concentrator Equipment Requirements Summary Table

Item	Description	Size/Capacity	No. Required	Power Installed
				kW
Compressors	General & instrument air	1,285 m^3/h @ 8 bar	2	300
	HP filter air	2,700 m^3/h @ 20 bar	2	360
Water treatment	Filtration	340 m^3/h	1	–
	Treatment (potable)	2 m^3/h	1	2
Crushers	Secondary cone	740 t/h	1	550
	Tertiary cone	1145 t/h	2	1,100
Mills	Primary	7.62 m dia X 12.2 m EGL	1	12,500
	Secondary	5.5 m dia X 8.2 m EGL	1	5,500
	Concentrate regrind	SMD-1100-E	2	2,200
Cyclones	Primary cluster	2,800 m^3/h	1	–
	Secondary cluster	2,175 m^3/h	1	–
	Concentrate regrind cluster	1,610 m^3/h / cluster	2	–
Blowers	Flotation cells	GM150S	3	555
Flotation cells	Rougher	200 m^3	7	1,575
(includes agitators)	Scavenger	150 m^3	6	1,110
	Rougher cleaner	70 m^3	6	660
	Rougher scavenger cleaner	70 m^3	5	550
	Rougher recleaner	70 m^3	5	550
	Scavenger cleaner	70 m^3	5	550
	Scavenger recleaner	20 m^3	5	225
Thickeners	Concentrate	24 m dia.	1	5.5
	Tailings	50 m dia.	1	37
Filters	Concentrate	62 t/h	2	264
Pumps	Various	0.033 - 2,500 m^3/h	111	6,870

Note that some equipment in the flotation section was specified for capital estimation before the most recent flotation results have become available. This includes the 2 x SMD-1100-E units which now need to be substituted for the IsaMills discussed in the text. Other equipment affected includes the secondary mill, thickener sizes and flotation cell numbers and types which have varied in accordance with the flotation test outcomes.

Table 17-4 is the projected equipment requirement overview for the proposed smelter.

Table 17-4: Smelter Equipment Requirement Summary Table

Item	Description	Size/Capacity	No. Required	Power Installed
				kW
Water	Cooling	8,000 m3/h	1	160
Oxygen plant	cryogenic	400 t/day	1	7,200
Drier	Steam	90 t/h	1	132
Crusher	Roll	10 t/h	1	15
Compressor	General plant air	2,850 m^3/h @ 6 bar	1	300
Furnaces	Direct blister	90 t/h	1	2,310
	Slag cleaning	330 t/batch	1	5,770
	Co recovery		1	3,850
	Anode		2	1,280
Waste heat boiler		30,000 Nm^3/h	1	–
Precipitator	Electrostatic	33,000 Nm^3/h	1	200
Bag filter		160,000 Nm^3/h	1	500
Casting wheel		50 t/h	1	–
Acid plant	Double contact	56,000 Nm^3/h dry gas	1	2,250

Table 17-5 is the estimated projected water, consumables and power requirements for the concentrator.

Table 17-5: Projected Concentrator Water, Power, and Consumables

Item	Description	Requirement
Power	electric	199,920 MWh/year
Water	raw make-up	144 m3/h
Reagents	Frother	375 t/year
	Collector (DTP)	275 t/year
	Collector (SiBX)	475 t/year
	Promoter (DTC1)	125 t/year
	Promoter (DTC2)	25 t/year
	Flocculant	250 t/year
Consumables	Grinding media	6,760 t/year

Table 17-6 is the estimated projected water, consumables and power requirements for the smelter.

Table 17-6: Projected Smelter Water, Power, and Consumables

Item	Description	Requirement
Power	Electric	120,500 MWh/year
Water	Raw make-up	163 m^3/h
Consumables	LFO	11,655 t/year
	Coke	18,633 t/year
	Refractory Materials	720 t/year
	Limestone	39,000 t/year
	Technical Oxygen	105,140 t/year
	Electrode Paste	550 t/year

17.5 Comments on Section 17

The concentrator flowsheet is currently configured without recycle streams on the same basis as the XPS flotation testing flowsheet. This arrangement is unusual and may result in difficulties controlling final concentrate grade in the face of varying feed characteristics. Modern flotation flowsheets typically reject cleaner scavenger tailings but rarely reject streams from the recleaner stages as is shown in the concentrator flowsheet.

Although the no-recycle flowsheet has been successful at laboratory level on a limited sample set it may be found in future work to have potential for either excessive tails copper losses or excessive silica capture into final concentrates.

Exploration of flowsheet options incorporating recycles will be conducted as part of the next study phase.

Currently the concentrator feed discussion does not differentiate between supergene and hypogene feedstocks and it is essential that this is detailed in the next study phase. Supergene copper concentrate is expected to be poorer in sulphur than hypogene generated concentrate and the proportion of each in the feed will have a significant bearing on flash furnace operation and acid production.

The equipment listed in Table 17-3 differs from the equipment described in Section 17-3. The main differences are a change of regrind mill type and in regrind milling power, a decrease in concentrate thickener diameter and an increase in tailings thickener diameter. In the context of the project estimate, and in the opinion of the AMEC QP, these differences are not material.

It must be recognised that cobalt recovery is included in the capital (the CoRF) but not in the revenue stream or the subsequent financial analysis.

It must also be noted that the smelter component descriptions are based on a typical package proposal from a vendor and may change in the next design phase as specific characteristics of the Kamoa concentrates and concentrate production schedule drive design outcomes.

18.0 PROJECT INFRASTRUCTURE

18.1 Introduction

This project infrastructure section incorporates assumptions, analysis and findings of the 2012 PEA.

The information relevant to the project infrastructure supporting the preliminary mine plan prepared during the 2012 PEA is included in this section and has not been updated because the QPs consider that the 2012 PEA is still current, given the assumptions supporting the outcomes remain reasonable. The effective date of the 2012 PEA results remains 5 September 2012.

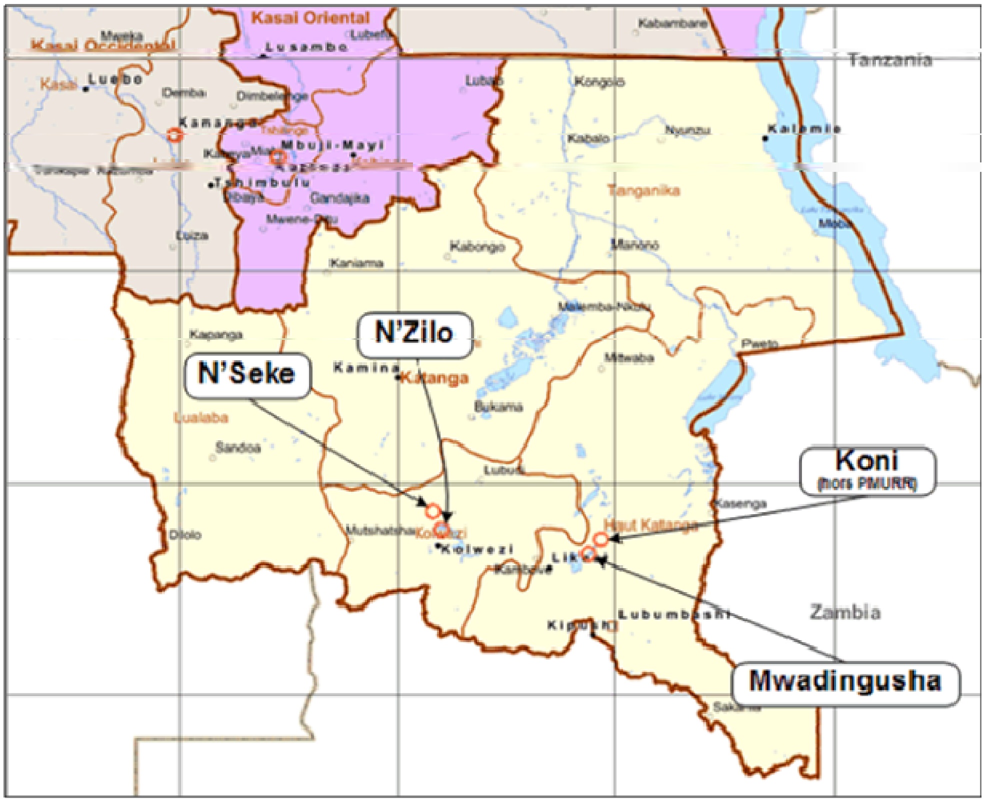
18.2 Power

18.2.1 Generation

As described below, the Project is eligible, after having signed in June 2011 a Memorandum of Understanding (MOU) with SNEL, to rehabilitate two existing hydro power plants of 113 MW aggregate generation capacity. After full rehabilitation, the Koni and Mwadingusha power plants could provide the Project with up to 80 MW of power. The cost of the rehabilitation will be financed by Ivanplats through a loan to SNEL. The loan will be repaid by SNEL through a deduction from the Company's monthly power bills incurred over the life of the mine. The MOU contemplates that following such an upgrade, Ivanplats would have an entitlement of up to 100 MW from those facilities, which the Company believes to be sufficient for the infrastructure contemplated in the current mine plan. The MOU led to the signing of a pre-financing agreement with SNEL for rehabilitation works on the Mwadingusha power plant, in June 2012. This pre-financing agreement stipulates the Company's exclusivity to conduct full rehabilitation on both plants. Financing, administration, operation and maintenance agreements for the full scope of rehabilitation have been settled but are currently being finalized. Ivanplats have advised that the agreements with SNEL will provide for 100 MW of power to the Project. Ivanplats has also signed an MOU with the Ministry of Energy in order to reserve a site for building a new greenfield power plant that could cater in case of need, for power supply for any future production capacity increase at the Project.

Figure 18-1 shows the locations of the existing power plants in relation to the Kamoa site. Figure 18-2 shows the state of electrical powerline infrastructure in proximity to the Project area.

Figure 18-1: Map showing Koni and Mwadingusha Power Plants in relation to Kamoa Site



Note: Figure generated 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 18-2: Example, Existing Power Transmission Lines in Proximity to Kamoa Site

Note: Photograph courtesy Ivanplats, 2012

Mwadingusha Hydroelectric Power Plant

The Mwadingusha (M'usha) hydro power plant is located on the Lufira River, approximately 70 km from the city of Likashi in the district of Katanga in the DRC. The hydro facility was built in 1928 and comprises some six turbines with an installed generation capacity of 71 MW at a gross hydrostatic head of 114 m. Turbines 4 and 5 were installed in 1938, whilst turbine 6 was installed in 1953. Of the turbines installed, only turbines 4, 5, and 6 are currently operational.

Koni Hydroelectric Power Plant

Koni is located 7 km upstream of M'usha 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 turbine 2 is currently operational.

Greenfield Hydro Site

In order to meet the Project's long term power requirements, which could reach 250 MW, a greenfields hydroelectric facility may also be required. In November 2011, Ivanplats signed an MOU with the Ministry of Energy of DRC to conduct an identification of the most suitable greenfields site amongst five potential sites already designated by the ministry. Ivanplats plans to conduct a due diligence and a conceptual assessment in this regard. Such a due diligence is likely to be followed by more detailed studies for the chosen site(s).

18.2.2 Transmission and Substations

Three existing 120 kV transmission lines deliver power to SNEL's transmission infrastructure through the RO and Shilatembo substations as detailed below.

- 120 kV transmission line between Koni and M'usha.
- 120 kV transmission line between M'usha and RO substation.
- 120 KV transmission line between M'usha and Shilatembo substation.

In order to achieve high power availability over the longer-term a new double 220 kV circuit transmission line will be required. However in the interim, 80 MW can be supplied to the Project over a new 10 km transmission line from the SCK substation in Kolwezi to a new 220/33 kV substation at Kamoa. A high level design was conducted by Hatch Limited (Hatch) in 2012, which included three phases for this new substation. Ivanplats advised that the current plan is to have 10 MW of construction power available on site by Q1 of 2014. Erection of the new power lines to provide this construction power is planned to start in H2 of 2013. Phase 2 of the power supply will be the supply of 100MW operational power, planned for Q1 of 2017. These plans are not final as further assessment for power transmission will depend as well on the greenfield hydro power location and capacity.

18.3 Tailing Dam

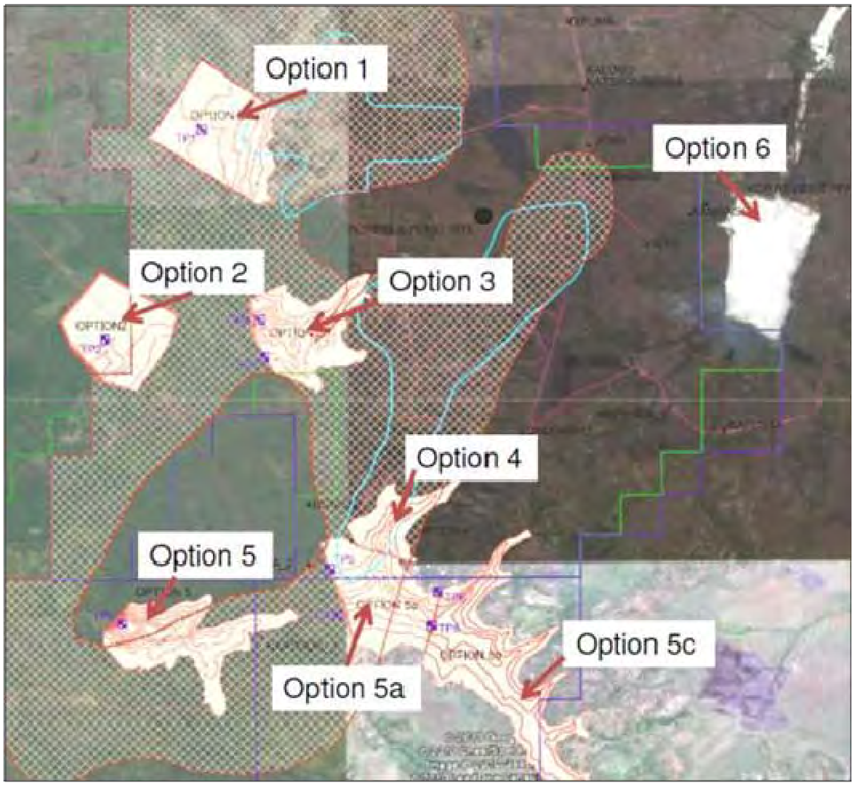
The tailings dam conceptual study focuses on the initial phase of mining and addresses the various options for a surface Tailings Management Facility (TMF) by comparatively appraising each site.

A total of 300 Mt of mineralized material from mining operations will be treated over the LoM with over 95% being discharged as tailings.

Various tailings disposal methods were compared, namely filter cake, paste or thickened tailings and tailings slurry. A scoring system for a number of factors associated with tailings disposal was adopted to enable a ranking of the three options. It was concluded that disposal using the slurry method is the preferred option for this site.

Eight potential TMF sites have consequently been identified by AMEC and formally assessed with respect to the social, environmental and technical characteristics (Figure 18-3). Two sites were eliminated from detailed consideration, namely Option 4 and Option 6. A scoring system was adopted to facilitate the ranking of the remaining six sites.

Figure 18-3: Tailings Dam



Note: Figure generated by AMEC, 2012. Map north is to top of plan. Width represented across figure is approximately 20 km.

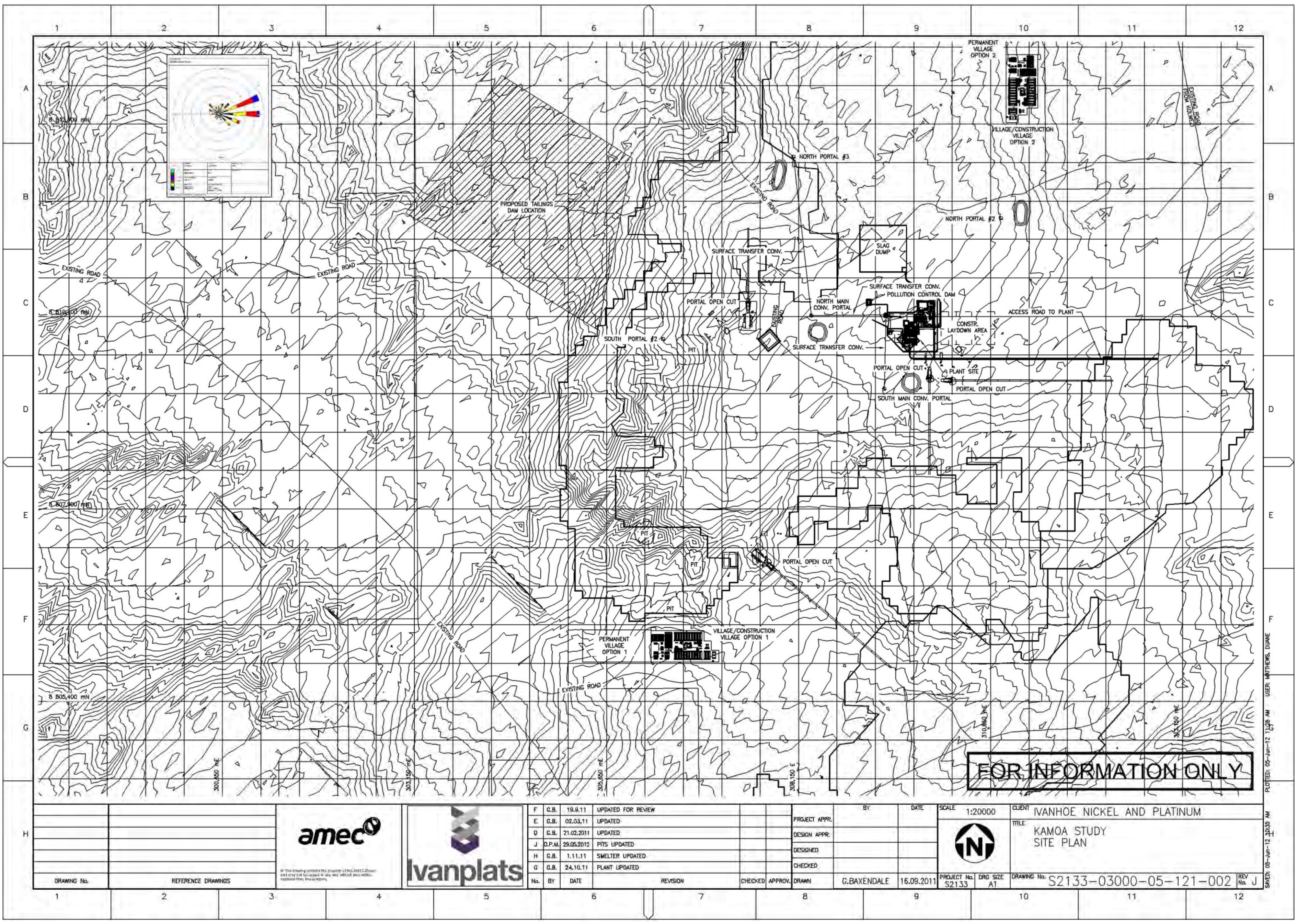
The results of the socio-environmental and technical assessments revealed that TMF Options 1 and 2 have equal top ranking with TMF Option 3 in third place. TMF Options 1 and 2 are very similar in most aspects with particular emphasis on the fact that both sites will probably require a high-density polyethylene (HDPE) liner. TMF Option 3 shows characteristics that indicate an HDPE liner will not be required. TMF Option 3 is located within the environs of low-grade mineralization on the boundaries of a planned open pit, which therefore eliminates this option.

Based on the above and the analysis undertaken, it is recommended that a Pre-feasibility study be undertaken for an HDPE-lined facility at the TMF Option 1 location.

18.4 Site Map

A site map is provided in Figure 18-4 that displays the locations of the proposed plant site, open pits and underground mining location, waste rock storage facilities and the tailings facility.

Figure 18-4: Site Conceptual Infrastructure Layout Plan



18.5 Communications

Current site communications comprise satellite phones and a VSAT internet link. Cell-phone service is available in Kolwezi.

18.6 Logistics

18.6.1 Overview

AMEC Minproc was tasked with undertaking a preliminary construction and operational logistics assessment for the Project in October 2010 (AMEC Minproc, 2010). This desktop study report is referenced. Findings are summarised below.

The Project is 25 km from Kolwezi and accessible by existing dirt and tar roads. The roads to Kolwezi from the relevant border posts are in a fair to poor condition. The ability to transit these roads easily is seasonal.

The DRC is a land-locked country and goods from outside of Africa are generally transported by sea to: Richards Bay and Durban in South Africa; Walvis Bay in Namibia, Beira and Maputo in Mozambique and Dar es Salaam in Tanzania and transported via one of a number of road and/or rail routes.

It is important to note that there are a number of techno-economic barriers to utilising the rail lines in each of these countries and thus, road transport is currently the most economic and preferred method for moving materials to and from site for both the construction and operational phases of the project.

The recommended routes for the transportation of materials are from and to either:

- Durban (South Africa) to Kamoa (DRC) (Route 1)
  - Durban/Johannesburg via Zimbabwe/Zambia/Chirundu Zambia border to the Kasumbulesa DRC border to Lubumbashi/Likasi/Kolwezi/Kamoa site.
  - Transit time is approx 14 to 15 days.
  - The maximum length and corresponding maximum weight of a truck and twin trailer unit in South Africa is 22 m and 56 tonnes respectively (normal loads).
  - SDV reports that up to 385 tonne net payloads have been transported in South Africa under an "abnormal load classification". Height and width dimensions are restrictive but are allowed with special permits, escorts, and modifications.

  - Maximum weight restriction on Lualaba bridge close to Kolwezi – 45 tonnes, alternatively a ferry may be used maximum gross weight 90 – 100 tonnes. SDV reports that they are currently transporting payloads of up to 80 tonnes to Kolwezi by road.

- Durban (South Africa) to Kamoa (DRC) (Route 2)
  - Durban/Johannesburg via Martinsdrift or Lobatse through Botswana and into Zambia via the border utilising the Kazungula ferry can also be utilised via Livingstone/Lusaka/Kitwe/Chingola/Kasumbalesa/Lubumbashi/Kamoa.
  - Transit time is approximately 13 to 15 days.
- Dar es Salaam (Tanzania) to Kamoa (DRC)
  - Dar es Salaam/Tunduma border – Kitwe/Kasumbalesa/Lubumbashi/Likasi/Kolwezi/Kamoa.
  - Transit time is approx 11–12 days.

From the above, from transport logistics and duration point of view, it can be seen that Dar es Salaam is a preferred port and transport route for goods sourced from and exported to locations outside of Africa.

However, if the Katanga – Benguela line (a branch of the Katanga Railway linking the Tenke junction just north-west of Likasi via Kolwezi to Dilolo at the Angolan border and Luau to the Atlantic port of Lobito) is re-instated, then Lobito would be the port of choice and rail the preferred transport method, as it was for the DRC and Zambian copper producers before the Angolan civil war of the 1970s. Currently, the line between Lobito and Dilolo (1,344 km) and the associated rolling stock is being upgraded by the Angolan and Chinese Government. No movement has been made in the re-instatement of the Katanga line between Dilolo and Kolwezi a distance of approximately 350 km (line of sight). AMBL has started contact with the Société Nationale des Chemins de Fer du Congo (SNCC) for potential railway prospects. Further assessments are needed in this regard.

A rail siding could be established on-site, by creating a new spur from the Luilu rail siding to Kamoa, alternatively, materials could be transported to and from Kamoa to this siding by road. Figure 18-5 shows the state of the existing laid track near the Kamoa Project site.

Figure 18-5: Existing Rail Track

Note: Figure courtesy Ivanplats, 2012

Where materials are required urgently, these will be air freighted to either: Kolwezi, Lubumbashi, Lusaka or Ndola airports.

No major issues are expected in the movement of goods and materials to and from site. However it is recommended that:

- The applicable protocols are implemented to allow goods to move on a duty free basis between countries of supply and or transit.
- A logistics company be appointed and central warehousing facilities set up to consolidate transport loads and ensure that bonds are not retained on shipping containers.

18.7 Airports

Lubumbashi International Airport is an airport in Lubumbashi, Democratic Republic of the Congo (IATA: FBM, ICAO: FZQA). The airport 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 serviced with the following airlines on a regular basis: 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.

Kolwezi Airport (IATA: KWZ, ICAO: FZQM) is a local airport serving Kolwezi, the capital city of the Lualaba District of the Katange Province in the Democratic Republic of Congo. The 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 above two airports will be utilised to transport people, goods and material to the project site during the life cycle of the project and during the operations phase.

18.8 Considerations

18.8.1 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.

18.8.2 Maintenance

There are no commercial type break down facilities between Chingola and Lubumbashi; however, there are a few commercially owned breakdown rigs with a towing capacity of up to 30 tonnes.

18.8.3 Power Lines

Power lines crossing roads are evident in places and the impact thereof on the transport of abnormal loads needs to be investigated.

18.8.4 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.

18.8.5 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.

18.8.6 Carbon Tax

The South Africa Government is currently considering the introduction of carbon taxes which may influence Logistics preferences. As details are not available, consideration of implications will happen in future study phases.

18.8.7 Logistics Companies

The growth of mining in the DRC is placing increasing pressure on local logistics companies and the supporting infrastructure. On this basis consideration should be given to establishing a logistics company in the DRC either directly or as a joint venture with other mining companies and/or a local or international logistics company. The mining companies may need to provide capital for fixed and mobile logistics infrastructure. Large logistics companies able to provide a comprehensive logistics service to the Project include, but are not necessarily limited to:

- SDV Transami part/Bollore Group (www.sdv.com or www.bollore.com)
- Unidel Carriers (www.unidel.co.za)
- Polytra (www.polytra.be).

18.8.8 Operational Logistics

Reagents and Consumables

During the operational phase, all reagents and consumables should be sourced though and transported from South Africa, unless suitable reagents and or consumables can be sourced locally and or in neighbouring countries. Reagents and consumables procured from outside of South Africa should be moved through Dar es Salam unless other routes should be found more economically optimal.

The routing of reagents and consumables to Kamoa will be the subject of a future, separate, transport study.

18.9 Water and Wastewater Systems

18.9.1 Bulk Water

Process water for any planned mining operation could be obtained from open-pit or underground water collection, recycling of process water, water management ponds and from re-treatment of water from waste piles. During additional Project advancement studies, appropriate sources of process water would be identified.

The bulk water demand has been estimated at ±570 m^3 per hour for the 5 Mtpa case (93.1% availability). Water storage assumes a raw water dam and a storm water dam.

The process plant design has a fire water tank, gland service water tank and a tank for process water reticulation.

18.9.2 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 is likely to 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.

It is expected that 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. An appropriate drinking water standard will be applied to indicators such as bacterial content, residual chlorine, turbidity and dissolved solids.

18.9.3 Potable Water Reticulation

Potable water will be distributed via pipe racks and sleeper ways along with other services where possible and underground as necessary.

18.9.4 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.

18.10 Fire Protection and Detection

The water supply for firefighting will be capable of providing the required firewater flows for any combination of hydrant-monitors, sprinkler systems, standpipe systems, deluge systems and responding fire apparatus based on a single fire event. The firewater distribution system shall be designed to supply the specified rate to each fire hazardous area. The firewater system, including connections to the system, will be designed to provide a high degree of reliability. Connections to the firewater system for any other service are not acceptable.

Fire hydrants shall be located when the plot plan has been finalized. Hydrants shall be "industrial standard" (not "commercial standard"). Hydrants shall be located on the roadside of all pipelines and drainage ditches and shall not be located within diked areas for tanks. Hydrants shall be located within 6 m of the roadway.

Hose reels should be placed on each equipment structure platform level and distributed inside process buildings wherever fire hazardous equipment is located. At least one hose reel is required per 1,000 m^2 of plant area.

Deluge systems shall be considered for:

- Pumps handling flammable liquids
- Lube oil consoles for critical equipment
- Transformer Bays
- Flammable Liquid Storage areas
- All other hazardous materials areas.

Placement of hand extinguishers shall meet prescribed specifications. Hand extinguishers to be placed in fire hazardous process areas, non-fire hazardous process areas, electrical rooms, non-process buildings, control rooms, and all company vehicles.

Fire detection equipment shall include a Fire Indicator Panel (FIP) located in the Main Control Room area and local intelligent units or Sub Fire Indicator Panels (SFIP) as required located around the site. The total control and indicating equipment shall be arranged as a distributed system preferably interconnected by a single communications and power loop on a loop in loop out basis at each device to minimise site wiring requirements.

The Fire Detection System will be independent of the Process Control System (PCS) and shall be specified as part of the overall Fire Protection System for the plant, which shall include the Fire Water System and Inert Gas Suppression Systems.

18.11 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-pat employees will be provided by an outside contracting service.

18.12 Buildings

The following buildings will form part of the project infrastructure:

18.12.1 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

- Messing and canteen
- Satellite ablutions.

18.12.2 Structural Steel Buildings

- Plant workshop (light crane loads)
- Plant store.

18.12.3 Additional Building

- Explosive Storage.

18.13 Permanent Housing

18.13.1 Accommodation

A permanent village will be constructed to provide accommodation as follows:

- Three-bedroom family units for middle management
- Four-bedroom family units for senior management
- Single units for unmarried staff
- Single units for contractors.

18.13.2 Facilities

The following facilities will be included:

- Canteen
- Recreation room
- Swimming pool
- Sports facilities
- Fire trailer
- Gate house and bus loading area
- Refuse removal.

18.13.3 Roads and Services

The following roads and services have been provided:

- Perimeter security fence
- Blacktop roads designed to an appropriate residential standard
- Parking
- Stormwater drainage
- Water reticulation, sized for fire flows and provided with hydrants
- Sewer reticulation
- Sewage treatment
- Mast lighting
- Transmitters and receivers for mobile telephones
- Internal and external communications.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Introduction

This marketing section incorporates assumptions and analysis used in the 2012 PEA.

The work prepared for Section 19 Marketing of the 2012 PEA has not been updated. It is considered that this work remains current as there has not been any additional work or changes to the assumptions that are likely to result in a material change to the conclusions of Section 19 in the 2012 PEA.

The QP for Section 19 of this Report is Bernard Peters.

The effective date of the 2012 PEA results remains 5 September 2012.

The Kamoa copper project is assumed to start production in 2017. The base case considers a 5 Mtpa mine producing blister copper and sulphuric acid as final saleable products.

To date Ivanplats has not advanced contract and market studies, apart from initial analysis of blister copper, acid and the potential sale of concentrate to Zambian smelters as an alternative to direct construction of a smelter.

Based on the analysis a long term Cu price of US$2.85/lb was used and a long term acid by-product price of US$250/t acid was used for the base case. These would appear to be reasonable relative to the prices used on other studies.

19.2 Blister Charges

The following are indications of the present blister commercial terms in the African market. The terms are usually negotiated on an annual basis.

19.2.1 Payments

Copper Payment – after a deduction of 0.3 units, the buyer shall pay the seller the agreed copper content at the official LME (London Metal Exchange) cash settlement quotation for Grade 'A' copper, averaged over the quotational period.

19.2.2 Deductions

Refining Charges – the buyer and seller will also negotiate a blister refining charge expressed in a rate per dry metric tonne. The present annual refining charge is between US$90 and US$130/t. A charge of US$100/t has been selected for the economic analysis.

It is common for the buyer to arrange for the transportation and to charge the cost back to the seller. The present inland charge to the port of Durban is approximately US$380/t. The transportation charges are highly volatile.

19.3 Sulphuric Acid Credit

The Congolese copper belt is a net acid consuming area. The majority of the copper and cobalt in the area is in the form of copper or cobalt oxides, and a leach SX/EW process is utilized to produce final product. This process is acid consuming and a number of the operations on the copper belt operate sulphur burning acid plants to produce acid, others purchase acid from Zambia or from overseas via the ports such as Walvis Bay or Durban. The sulphuric acid price is rather volatile, operators in the DRC are reported as currently paying US$250/t to US$480/t of acid delivered from Zambia and some report the cost from Namibia as over US$600/t. A recent published long term operating cost for acid produced from an onsite acid plant in the DRC is US$206/t. Long term sales prices may include some additional above operating but below total cost, including capital, for an acid plant. For the purpose of the study a long term acid credit of US$250/t has been used. AMEC recommends that Ivanplats undertake detailed market analysis of the acid demand, supply and pricing, in particular in the Katanga region for the further studies to confirm the acid price assumptions.

19.4 Concentrate Sale

The base case scenario assumes the production of blister copper for sale in Asia and sulphuric acid for local sale.

A visit to three Zambian copper smelters was undertaken on 21 and 22 November 2011 by Jay Gow, VP Marketing Ivanplats, Steve Amos, VP Metallurgy Ivanplats and Bronwyn Swartz, Senior Process Engineer, Ivanplats. The purpose of the visit was to determine if any potential exists to export copper concentrate to the Zambian copper belt for toll smelting.

The Kamoa copper project is expected to produce a copper concentrate grading approximately 33% copper and no payable precious metals. Initial indications are the three smelters require additional concentrate feed. However if this option was followed the challenges will be: (1) gaining approval from the DRC government to export copper concentrate and (2) the cost and logistics associated with transporting copper concentrate from Kamoa to the smelters. For these reasons the Ivanplats has selected an on-site smelter option for the PEA.

19.5 Mopani Copper Mines (MCM) – 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 300,000 tpa copper cathode. They do not produce enough concentrate from their own mines and therefore must purchase or toll concentrate from third parties. Brook Hunt has reported that MCM produced about 220,000 tpa of cathode in 2010. MCM have recently announced they will build another shaft to access additional ore because the existing mining operation will end in 2017.

19.6 Chambishi Copper Smelter Limited (CCS)

CCS is owned 60% by Yunnan Copper and 40% by China Nonferrous Metal Mining Company (CNMC), the smelter began operation in 2009. The smelter is located about 30 km east of Chingola. The ISASMELT process produces 150,000 tpa of blister copper; they do not have a refinery. 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/customers in China, Korea, Germany, and India.

CSC plans to increase the smelter production to 250,000 tpa but is dependent on copper concentrate supply. CSC is a good candidate to purchase Kamoa concentrate.

19.7 Konkola Copper Mines plc (KCM)

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 300,000 tpa using the Outotec Direct to Blister Technology. KCM is presently producing at 240,000 tpa. Their own mines produce about 50% of their feed and the balance is purchased from Barrick (Lumwana) and First Quantum (Kansanshi). The final product (blister and cathode) is shipped to their rod plant in Dubai and to customers in China.

KCM would be a possibility for Kamoa copper concentrate.

19.8 Copper Concentrate Treatment and Refining Charges (TC/RC)

If Kamoa were to produce a copper concentrate for the first couple of years of production, they would need to negotiate with the receiving smelters the annual commercial terms. The present TC/RC's for the Zambian smelters reflect the difficultly and expense of shipping copper concentrate offshore.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

This section on environmental, permitting and social and community impacts incorporates assumptions, analysis and findings related to the 2012 PEA.

The information relevant to the environmental, permitting and social and community impacts supporting the financial analysis prepared during the 2012 PEA is included in this section and has not been updated because the QPs consider that the 2012 PEA is still current, given the assumptions supporting the outcomes remain reasonable. The effective date of the 2012 PEA results remains 5 September 2012.

African Mining Consultants Ltd was commissioned in early 2010 to conduct a DRC regulatory Environmental Baseline Study (EBS) and corresponding report. The report discusses the findings of an EBS conducted for the Project from February 2010 to February 2011.

African Mining Consultants is an environmental and mining engineering firm based in Kitwe, Zambia. The company was formed in 1994 following the signing of affiliation agreements with Golder Associates, and was registered in 2004 as a Congolese environmental consulting firm. This registration has been recently updated. African Mining Consultants has developed particular expertise in the practice of applied environmental engineering, particularly relating to base metal processing, in addition to open pit and underground mining project expertise, environmental audits, impact assessments and the development of environmental management plans.

The EBS report presents the characterisation of the existing conditions related to environmental and social conditions in the Project area.

The report is divided into five sections:

- Introduction – provides the background of the Project, the main objective of the report, the approach, the Project developer, consulting firm, nature of the permit, the Project location and the regulatory framework.
- Project Description – discusses the geology of the area and the proposed mine infrastructure.
- Physical Environment – presents the existing biophysical environment in the Project area, such as climatic conditions, topography, soils, risks and natural disasters, air quality, noise levels, hydrology and hydrogeology.

- Biological Environment – discusses the terrestrial flora and fauna and the aquatic biodiversity of the area.
- Social Environment and Cultural Heritage – this section describes the social environment of the area in terms of demography, livelihood and infrastructure, and archaeological aspects of the area.

20.2 Project Description

The exploration activities have been conducted in the permits PR702, PR703, PR704, PR705, and PR706 east of Kolwezi in the mineral-rich Katanga Province of the DRC.

The Project area is currently located in PR705, from where the copper will be mined and processed.

The Project area has been explored since 2004. Initial significant copper in soil anomalies in around the Kamoa area and Makalu domes called for follow-up exploration.

Reverse circulation drilling in 2006 and diamond core drilling in 2008 revealed potentially economic copper within a regionally favourable horizon extending over at least 130 km^2.

This EBS will culminate in the development of an Environmental Impact Study (EIS) including an Environmental Management Plan (EMP), which will be submitted to DRC authorities for consideration. Upon approval, the DRC authorities will issue an exploitation permit to AMBL. The Exploitation Permit was granted on 20 August 2012.

The Project is envisaged to be an underground copper mine with small open pits providing initial plant feed.

The underground operations will be constructed to the required depths over the life of the mine by standard underground mine development strategies such as room-and-pillar mining.

The mine will require portal and exploration decline developments, ore and waste conveyor systems, run-of-mine (ROM) pads, a waste disposal area, primary and secondary crushing stations and associated auxiliary production facilities including a concentrator and smelter.

Living quarters will be constructed for employees and access roads will be built and/or upgraded to meet the traffic and transport requirements for the mine site.

Mined plant feed, sub-economic grade plant feed and waste overburden will be separated and transported for subsequent extraction and storage respectively. Copper in the plant feed will be extracted using concentrator and smelter. The copper concentrate will be sent to an on-site recovery facility to produce the finished product containing approximately 99% copper as a minimum.

20.3 Physical Environment

20.3.1 Climate

- The region experiences rain from September to May, with drier months from June to August.
- December through February are the wettest months with December generally having the highest rainfall.
- The Project area appears to receive higher levels of rainfall when compared to Kolwezi Airport's average monthly rainfall.
- Rainfall data collected at Kaponda and Kalundu rain gauge stations suggest high local rainfall variation in the Project area.
- Kamoa annual maximum and minimum temperature ranges are higher and lower than the Kolwezi Airport temperatures respectively.
- Evaporation generally exceeds precipitation for most of the year with potential evaporation highest in the driest months.
- Average monthly evaporation at Kolwezi Airport ranges from 5.9 mm to 9.7 mm in warmer months, and ranges from 1.7 mm to 10.2 mm from 2004 – 2010.
- The lowest monthly evaporation rates occur in the wet season (November to January).
- Mean monthly humidity levels at Kamoa varied from a minimum of 39% (August) to a maximum of 89% (January) between May 2010 and March 2011.
- The dominant wind direction recorded at the Kamoa Project weather station is from a north-easterly to easterly direction, with average wind speeds of 0.9 m/s, gusting up to 3.5 m/s.

20.4 Topography

- The site lies at the edge of a north-north-east to south-south-west trending ridge which is incised by numerous poorly to well developed drainages.

- The Project is located on the crest of a low lying ridge, as part of a series of undulating ridges with incised open valleys forming part of the Lualaba catchment.
- The elevation of the Project area ranges 1,300 m to 1,540 m 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 landscape is dominated by dambo (regional name for low-lying valley related wetlands) and dilungu (regional name for Kalahari sand plateau).

20.5 Soils

- The majority of the soils in the Kamoa area are Class V or VI or N1 and N2 i.e. the soils are unsuitable for agriculture.
- Limited areas will respond to remedial actions e.g. fertiliser, manure, but others will remain unsuitable permanently. The soil groups which fall into Unsuitable categories are Arenosols, Plinthosols, Ferralsols, Cambisols & Alisols.
- Arenosols & Plinthosols are unsuitable for agriculture due to water logging and soil texture. These soil groups cover vast areas of dambo wetlands and dilungus. Cambisols & Alisols, being located mostly on exposed ridges and slopes, have poor rooting conditions which make them unfit for successful agriculture.
- Management of land by the local populations has enabled some of the soils in the Project area to be classified as Class IV or S3 i.e. marginally suited to agriculture based on severe limitations which reduce productivity. Soil types in this classification are mostly Acrisols, Ferralsols and Plinthosols.
- Much of the Project area is under slash-and-burn agriculture and is left fallow for many years between phases of agricultural use.
- One fertile group of soils (Class III or S2 i.e. moderately suited to agriculture with land having moderate limitations) was sampled in pit P19. S2 group soils (Nitisols and Acrisols) are mostly found in undisturbed forests or on land which has been abandoned for long fallow periods to allow the fertility to regenerate.
- Existing and future impacts on soil quality are land use changes and disturbance, soil contamination, soil degradation and soil erosion.

20.6 Seismicity

- The seismic hazard assessment was described in AMEC (2010) and the key findings described below. The Seismic Hazard Map (United Nations Global Seismic Hazard Assessment Program (GSHAP)) and the United Nations Office for the Coordination of Humanitarian Affairs (OCHA) resource centre indicate that:
- The Kamoa Project is located in a zone of Moderate Intensity Seismicity with Peak Ground Accelerations up to 0.8 m/s2.
- This value is the equivalent of an Operating Basis Earthquake (OBE).
- The OBE is considered to occur not more than once in a period of 100 years and would be the basis for engineering and designing structures in the Kamoa area.
- The OBE would be refined based on further site-specific studies. AMEC have concluded and shown, based on further analysis in the report above, that:
  - The region of the Project area is potentially seismically moderately to strongly active.
  - This seismicity is associated with the Western Branch of the Western Rift Valley of Africa, specifically the Upemba and Mwelu graben extensions.
  - The regional OBE is 0.08 g (0.8 m/s^2) and the regional Maximum Credible Event for design purposes was a conservative 0.11g (1.1 m/s^2).
  - Further detailed analysis is required to describe the values mentioned with more confidence and accuracy.

20.7 Air Quality

- The area currently does not have any major sources of sulphur dioxide (SO2), ozone (O3) and nitrogen dioxide (NO2), and those sources that are present are unlikely to change significantly on a daily basis, as would be the case in an industrialised urban environment.
- The concentration of NO2, O3, SO2 and particulate matter (dust) sampled from April to December 2010 are well below the respective DRC limit values.
- There are marked cool dry season and wet warm seasonal effects on the concentrations of NO2, O3, SO2 and particulate matter (dust) measured at the Project site.
- Dust levels are highest in the cool, dry season months of July to September. Concentrations of SO2 and NO2 are highest in October and November.

- Further analysis of the site-specific conditions of AQ23 is required to determine the high dust levels measured in this area.
- There are potential impacts on air quality from outside the area e.g. Luilu Tailings Storage Facility and this should be further investigated and quantified.
- The continuation of the collection of air quality information is essential to determine a better understanding of the trends of the NO2, O3, SO2 and particulate matter (dust) in order to develop future models for impact prediction and assessment.

20.8 Noise

- There are three main sources of noise in the Project area, notably exploration machinery and vehicles, wind and rain, and local communities.
- Noise in the Project area (LAeq) is compliant with DRC guidelines for areas mostly under agricultural activities (70dBA) at all of the sampling points from February 2010 to January 2011, with values ranging from 48dBA to 65dBA (daytime) and 36dBA to 50dBA (night time).
- Intermittent noise (LAmax) is occasionally non-compliant with the DRC guidelines for areas under agricultural activity and exceeds 70dBA. These locations are usually villages but also Kamoa Camp (NS09) and the Project access barrier (NS10).
- There are no sensitive receptors such as dense housing, hospitals or schools or areas of commercial or industrial activities in the area.
- Ambient noise levels are highest around settlement areas.

20.9 Hydrology

- Surface water quality sampling was conducted by African Mining Consultants and a preliminary hydrological baseline study described in Golder Associates (2010a).
- Surface water sampling was conducted by African Mining Consultants from March 2010 to January 2011.
- All of the surface water samples were compliant with WHO Drinking Water Standards.
- The pH ranges across all sampling sites from 4.8 to 7.4.
- Insignificant concentrations of nitrite, manganese, aluminium, chromium, nickel, lead, zinc, cadmium, cobalt, arsenic, selenium or mercury were found in the surface water samples.

- All of the surface water monitoring sites contained copper concentrations, the highest being 0.92 mg/l, which was recorded at SW04 (Mukanga River).
- Fluoride in surface water samples occurred in June to August. It is recommended that the profile of this parameter be monitored over a full 12 months.
- Cyanide was sampled in SW 01 and SW 04 in March 2010. These levels have normalised since this date. The reason for this anomaly is thought to be either contamination or human activity (such as poison fishing or cassava fermentation) this cannot be confirmed.
- Natural sulphate concentrations in the surface water samples are lowest at the end of the rainy season and highest in October and November.
- All of the sites sampled were affected by seasonal flow changes which affected the sampled parameters through dilution and increased run-off. It is recommended that further sampling be continued and a site profile be generated for each surface water sample site based on sampled parameters.
- Golder Associates conducted a surface water monitoring program to assess flows, discharges, rainfall, catchment analysis and initiate the development of an Agricultural Catchments Research Unit (ACRU) model for the Project area.
- Eight sites were monitored for discharge with Lulua (downstream) being the highest and Kalundu (upstream) being the lowest.
- Decreasing stream water levels leads to increased water temperatures, with water temperatures in the sampled streams ranging from 16°C to 22°C.
- The main catchments in the Project area are Kamoa, Kalundu, Mukanga and Lulua. The largest catchment is Lulua.
- Discharge in the rivers increased at the start of the rainy season and decreased once precipitation ceased.

20.10 Hydrogeology and Groundwater Quality

- The hydrogeological study was conducted by Golder Associates and described in Golder (2010b). The report is quoted in this section.
- The geological and hydrogeological data gathered has provided an overview of the groundwater conditions across the Project area and enabled the baseline hydrogeological situation to be described.
- A localised primary aquifer is present within the Kalahari sands of the dilungus, which provides recharge to the underlying aquifer developed within the Upper Diamictite.

- The study has also confirmed that a widespread, well developed secondary, anisotropic aquifer is present within the hangingwall Upper Diamictite.
- The secondary aquifer is underlain by Pyritic Siltstone and Lower Diamictite, which are characterised by low permeability. They appear to act as an aquitard which separates the main aquifer from the deep footwall aquifer present in the Roan Sandstone.
- Information concerning the Roan Sandstone is limited at this stage, but the available data indicate that the piezometric water levels of the deep aquifer are some 30–50 m below the piezometric levels of the main hangingwall aquifer, confirming the presence of two separate aquifers.
- The Upper Diamictite comprises the main water-bearing horizon (main aquifer) of the area. This aquifer is well developed and with saturated thickness in excess of 200 m in places.
- The weathering thickness of the Upper Diamictite varies within a wide range from 20 to 60 m or more, and a minor aquifer is associated with the weathered zone.
- Groundwater occurrence within the main aquifer is controlled by the presence of fracture zones within the Upper Diamictite rock mass. Multiple fracture zones are usually present below ±50 m and since the entire sequence is saturated, the thicker the sequence, the higher borehole yields will be. This is illustrated by the variation in the cumulative blow yields obtained, which vary from 0.5 l/s and 10 l/s, with the higher yields generally associated with the thicker Upper Diamictite.
- Transmissivity values as determined from the limited testing undertaken vary from 3 to 250 m2/day with the average between 25 and 50 m2/d. These values confirm that the permeability of the Upper Diamictite is relatively well developed and groundwater inflow will require active management during mining.
- Water levels are generally shallow throughout the concession, between 10 and 20 metres below ground level, deepening to the west and north-west towards the West Scarp Fault zone.
- The groundwater flow in the main aquifer is towards the west and north-west.
- It appears that the West Scarp Fault acts as a discharge zone, capturing groundwater flow to the west. The convergence of piezometric heads of the main and deeper aquifers towards the fault zone, would support the finding that both aquifers discharge into the West Scarp Fault Zone.
- The water level in the sandstone of the Kamoa Dome anticline is below that of the main aquifer and it can be concluded that the main aquifer does not laterally recharge the sandstone. Furthermore, the footwall (sandstone) of the mineralized

zone on both sides of the dome is unsaturated above the piezometric level of the sandstone. This may mean that adjacent to the dome dewatering of the sandstone may not be required until the pits reach a depth of 80 or so mbgl – however, dewatering of the Upper Diamictite will still be necessary.

- The Kamoa Dome anticline acts as a barrier to groundwater flow, and forces groundwater flow in the Upper Diamictite aquifer to move around the dome to the south.
- The baseline assessment provides a well defined conceptual understanding of the general groundwater behaviour within the study area.
- An important finding of the study is that groundwater occurrence is widespread and relatively well developed. The fact that the main aquifer is fractured, possesses relatively high permeabilities and is formed in the hangingwall, means that proactive groundwater management will be necessary for mining. Dewatering will be needed. Further studies will confirm the requirement and forms of dewatering activities.
- African Mining Consultants collected and analysed groundwater samples from local village wells from March 2010 to January 2011.
- None of the groundwater samples contained measurable amounts of cyanide, nitrite, fluoride, manganese, aluminium, chromium, nickel, lead, zinc, cadmium, cobalt, arsenic, selenium or mercury in the monthly samples collected from March 2010 to January 2011. It is recommended that monitoring be maintained for these elements on a quarterly basis for at least another annual cycle before elimination from the sampling parameters.
- Copper concentrations in the sampled boreholes ranged from a minimum of zero (GW03) to a maximum of 0.66 mg/l (GW02). Seasonal effects appear to be slightly offset when compared to surface water patterns, with higher concentrations occurring later in the wet and dry seasons.
- pH levels vary in each borehole, generally, from pH 5.5 to pH 7.5. However GW01 experienced a pH of 2.4 when initially sampled in March 2010. This incident has not repeated itself in the borehole up to January 2011.
- Electrical conductivities in the groundwater samples are highest towards the end of the wet season (March 2010) and decrease throughout the sampled period. The highest conductivities were experienced in GW06 (378 µS/cm), GW05 (245 µS/cm) and GW04 (156 µS/cm).
- The total dissolved solids in the groundwater closely reflect the patterns of electrical conductivity. GW06 displayed the highest dissolved solids of 265 mg/l.

- Sulphates are present in groundwater with the highest concentrations being 46 mg/l at GW03, which is northeast of the Project area.
- It is recommended that further monitoring continue and the African Mining Consultants sampling sites are incorporated into the groundwater monitoring program, to monitor groundwater supplies of the local communities.

20.11 Biological Environment

20.11.1 Ecological Biodiversity

- The area does not contain any protected plants or areas listed in the DRC Regulations which are perceived to contain endangered species.
- In 2002, the total forest area (within the Greater Kamoa Concession area) was 50,636 ha, while total non-forest area was 30,812 ha. However, by 2010, the total forest area had been reduced to 49,972 ha while non-forest area had increased to 31,476 ha. The non-forest area includes built-up land, river flood plain areas or dambos, water, and roads within the Project area.
- The Project area is located along a plateau ridge that defines the western end of the Congo-Zambezi watershed area. It is characterised by wetlands (mainly covered with grass), forests and woodlands, which play an important role in the collection and recharge of rainwater into streams. These streams feed into the Lualaba, Kabompo and Zambezi drainage systems.
- The site is Neolithic and has been influenced by human habitation for a long period time.
- The area has important vegetation features that are useful for both ecological and biodiversity conservation. Although narrow in variation, the woody flora of the Project area supports a number of epiphytes. These include lichens, mosses, ferns, and orchids.
- Other plants of considerable conservation value include the tree fern Cyathea dregei, and Platycerium elephantotis. These are listed by the International Convention for the preservation of wild species from extinction, as plants liable to become very scarce due to changes in ecosystem conditions and overharvesting.
- Existing human activity (agriculture and charcoal) has had an effect on vegetation disturbance in the Project area.
- In April 2010, AMBL commenced a sustainable forestry project, along with the Project communities, under the management of Eco-livelihoods Ltd and Envirotrade Ltd, an organisation from the UK that specialises in ecologically

sustainable reforestation and land-use management programs. The pilot study is intended to run for 12 months and the results from this will lead to further studies and sustainable development programs.

- The area does not contain any totally protected animals or partially protected animals listed in the DRC regulations.
- There is low fauna biodiversity. This is due to the fact that the physical habitat is degraded through human activity such as deforestation for farming and charcoal production.
- The area does not contain any fish or aquatic flora and fauna species listed in the DRC Regulations which are perceived to be endangered species.
- Five sampling sites were investigated during the aquatic flora and fauna survey. Biological habitat assessments indicated that: three sites had optimal habitat conditions for aquatic species and two sites had sub-optimal conditions.
- Invertebrate Habitat Assessment (IHA) was conducted and indicated that of the three sites with optimal habitat, one site had a good river health classification, and the other two sites had adequate/fair river health; the two sites with sub-optimal habitats had poor river health based on a lack of sediment in the waterbed.
- A number of the recorded fish species may represent new species or subspecies of existing species (e.g. Kneria sp and Aplocheilichthys sp).
- A macro-invertebrate survey identified 26 different species in the Project area. The Ephemeroptera, Plecoptera and Trichoptera (EPT) analysis indicated the presence of these pollution-sensitive species at four of the five sampled sites.
- The potential impacts that the proposed mining activities may have upon these ecological values will be predicted and environmental management plans formulated after the final mine plan has been established.

20.12 Social Environment and Cultural Heritage

20.12.1 Heritage

The region is historically an important Neolithic area which has contributed to information on early settlements by Bantu people. However, there are no currently known heritage sites within the area of the Project.

20.12.2 Community Groups

There is a diverse mix of ethnic or tribal groupings with their own languages in the Katanga Province. The main languages spoken in the Province are Lunda, Kaonde, Swahili and French. The principal ethnic group in the Project area is the Lunda Tribe.

There are about 23 villages within and immediately surrounding the Project area with an estimated total population of about 20,000 (see Figure 4-1). Many of the villages are situated in the northeastern portion of Permit 705 and the southeastern portion of Permit 706 (outside the Project area), where the lithologies that host the Kamoa mineralization are absent due to erosion. Village populations are fluctuating upwards, as the general area has seen an influx of people looking for employment with exploration and drilling companies.

Typical village activities include charcoal burning, cultivation of cassava, sweet potato, and maize, and livestock tending of fowls, sheep, pigs, and goats.

20.12.3 Social Baseline

- The Democratic Republic of Congo is administratively divided into the city of Kinshasa and 10 provinces. These provinces are: Bandundu, Bas-Congo, Equateur, Kasai-Occidental, Kasai Oriental, Katanga, Maniema, Nord-Kivu, Province Orientale and South-Kivu. Kinshasa is the capital and the head office of national institutions. The capital can only be relocated to another location by way of referendum.
- The DRC is mainly dominated by two large provinces: which are Kinshasa and Katanga.
- The Project is located in the Mutshatsha Territory of the Kolwezi-Lualaba District in the Katanga Province.
- The Project area includes the two main traditional / cultural tribal Groupings of Musokantanda and Mwilu. These Groupings are composed of villages, which are directed by village Chiefs who rely on an elected Land Chief that reports to the Grouping Chief.
- The main ethnic groups in the region are the Ndembo (38%), the Sanga (32%), the Tshokwe (13%) and the Lunda (11%).
- There are 19 villages in the region affected by the Project with a total population of approximately 3,243 people.

- The average average household size is 6 persons (principally, the father, the mother and two or three children with other person such as a parent of the mother or father).
- The majority (88%) of the population live permanently in their villages and most of them (79%) are owners of the house where they live. However, Kaponda Village appears to be a relatively new establishment with a high number (92%) of families renting the houses where they live. The village seems to have developed based on the anticipation of the communities for work from AMBL.
- The population in the study area suffers from extreme poverty. The population is engaged mainly in agriculture (72.8%), but it is also engaged in other activities such as charcoal production and hunting. The population are also engaged in brick making and carpentry, and most have built their own houses.
- Approximately 87% of the population are Christian while the rest are animists or atheists.
- None of the studied villages have access to clean water (potable water).
- None of the studied villages benefit from continuous electricity supplies.
- There are few health facilities in the study area. The nearest clinic, a small health centre, is located in Walemba Village, located 20 km from the existing exploration camp. This clinic has limited medical supplies and little equipment.
- Since the expansion of the exploration campaign, it has been necessary to describe the Project area, especially to define in more detail the areas affected by the Project. It was proposed that these studies are part of the further work.

20.13 Surface and Groundwater

In February 2010, Golder Associates Africa (Golder) were requested to undertake a field visit to the Project to provide an overview of the surface and groundwater situation on-site.

The findings from the field visit were consolidated in the Golder report (PRO14626 of February 2010) entitled: “Site Visit Report and Scoping and PFS Level Programme for Hydrogeology, Geochemistry and Hydrology”. A report included an overview of the groundwater, surface water and geochemical considerations relevant to the intended mine development and a proposed programme to undertake the necessary baseline investigations for the concession.

Based on this documentation, AMEC appointed Golder on behalf of AMBL, to carry out the aforementioned Baseline Studies. These baseline studies have been completed and separate baseline reports for each discipline area have been developed and issued, namely: hydrogeology, hydrology, geochemistry and bulk water supply.

Key findings from these studies are summarised in bullet point below. Additional detail may be found in the relevant Golder Reports.

- There is sufficient water available to meet the projects water requirements within the Region.
- Whilst five plausible sources of water have been identified, these can be narrowed down to less than three in moving forward with the project.
- The meteorological conditions are unlikely to pose any significant project risk.
- The resource appears to be enriched in a number of what may be termed "hazardous/deleterious elements (Ag, Al, As, Au, Ba, Bi, Co, Cu, Fe, Mn, Mo, S, Sb, Se, Sr, Te, U and Zn)" and compounds (cyanide). The mobility of these elements/compounds in the ground water must be carefully managed during the mine's operational and closure phases, in order to address safety, health and environmental constraints and long term liabilities. Arsenic (As) is a very topical element in the mining industry and is toxic at very low concentrations.
- Waste rock, tailings and in situ material left behind in the mining process are likely to acidify the associated water and accentuate the mobility of certain hazardous elements. It is important to note that trace element mobility in groundwater need not be a function of pH. The management of acid mine drainage and the cost thereof, will need to be defined in detail as the project progresses.
- Given evidence of high dewatering rates in surrounding areas, there are some concerns that water management (pit and underground) may pose a significant, but as yet undefined cost to the project. This will only be quantified during the next project phase.
- Consideration needs to be given to all users of water (current and future) and the impact thereof of mining activities.
- To mitigate license to operate issues and long-term legal and financial liabilities, the water base line (quality and quantity) and the relative contribution of the Project and surrounding mines to future water quality and flows must be determined.
- The management of post closure water issues and the costs thereof, need to be defined in detail, in the next project phase.
- There are some concerns with the validity of water quality analytical data sourced from laboratories in Zambia. This will be addressed in subsequent project development phases.

21.0 CAPITAL AND OPERATING COSTS

21.1 Introduction

This section on cost estimation incorporates assumptions, analysis and estimates related to the 2012 PEA.

The costs prepared for Section 21 Capital and Operating Costs of the 2012 PEA have not been updated. It is considered that this work remains current as there has not been any additional work or changes to the assumptions that are likely to result in a material change to the conclusions of the work in Section 21 in the 2012 PEA.

The QPs for Section 21 of this Report are: Bernard Peters (Section 21.1, 21.2, 21.3, 21.7, 21.8), Mel Lawson (Section 21.4), and Dean David (Section 21.4, 21.5).

21.2 Capital and Operating Cost Summary

Capital and operating costs have been estimated separately for each of the key activities:

- Open pit mining
- Underground mining
- Concentrator
- Smelter
- General and Administration
- Owners Costs.

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 Payable Copper		
	LOM AVG	**First 5 Years Average**	**First 10 Years Average**
Mine Site Cash Cost	1.04	0.93	0.89
Realisation Cost	0.30	0.30	0.30
Total Cash Costs Before Credits	**1.34**	**1.22**	**1.19**
Acid Credits	0.24	0.24	0.24
Total Cash Costs After Credits	**1.10**	**0.99**	**0.95**

Table 21-2: Operating Costs

	US$M	US$/t Plant Feed Milled		
	Total LOM	**First 5 Years**	**First 10 Years**	**LOM AVG**
Site Operating Costs				
OP & UG Mining	10,476	28.92	32.20	34.99
Processing	3,125	10.55	10.45	10.44
Smelting	2,203	7.65	7.90	7.36
Tailings	83	0.27	0.27	0.28
General & Administration	1,616	6.87	5.63	5.40
Customs	342	1.01	1.09	1.14
Total	**17,844**	**55.28**	**57.54**	**59.60**
Operating Margin	**30,085**	**111.29**	**122.32**	**100.49**

The capital costs for the project are detailed in Table 21-3.

Table 21-3: Capital Investment Summary

US$M	Phase 1	Sustaining	Total
	Mining		
Open Pit Mining	63	5	67
Underground Mining	173	1,146	1,319
Subtotal	**235**	**1,151**	**1,386**
	Power & Smelter		
Smelter	313	38	351
Power	141	85	226
Subtotal	**455**	**122**	**577**
	Concentrator & Tailings		
Concentrator	270	12	281
Tailings	42	372	414
Subtotal	**312**	**383**	**696**
	Infrastructure		
Site Facilities and Temporary Works	234	141	375
Off-site Infrastructure	24	14	38
Subtotal	**257**	**155**	**413**
	Indirects		
Construction Indirects EPCM	155	2	157
Subtotal	**155**	**2**	**157**
	Owners Cost (incl. Drilling & Studies)		
Owners Cost	205	92	298
Closure		200	200
Subtotal	**205**	**292**	**498**
Capital Expenditure Before Contingency	**1,621**	**2,106**	**3,727**
Contingency	368	453	821
Capital Expenditure After Contingency	**1,989**	**2,560**	**4,549**

21.3 Open Pit Mining

The open pit costs were estimated using information received from two contractors with experience in the region and benchmarked using other information from operations in the DRC.

The costs and rates assumed are shown in Table 21-4 and Table 21-5.

Table 21-4: Mobilization, Demobilization, and On and Off-Site Overhead Costs

	Unit	Cost
Mobilization and Establishment of All Plant	US$M	7.2
Mobilization and Establishment of All Fixed Facilities	US$M	4.8
Demobilization of All Plant	US$M	3.6
Other Demobilization Charges	US$M	1.2

Table 21-5: Average Open Pit Mining Costs

Item	Cost (US$/t Rock)
Mine Capital	0.34
Operating	3.65
Total Open Pit	4.00

21.4 Underground Mining Cost Estimate

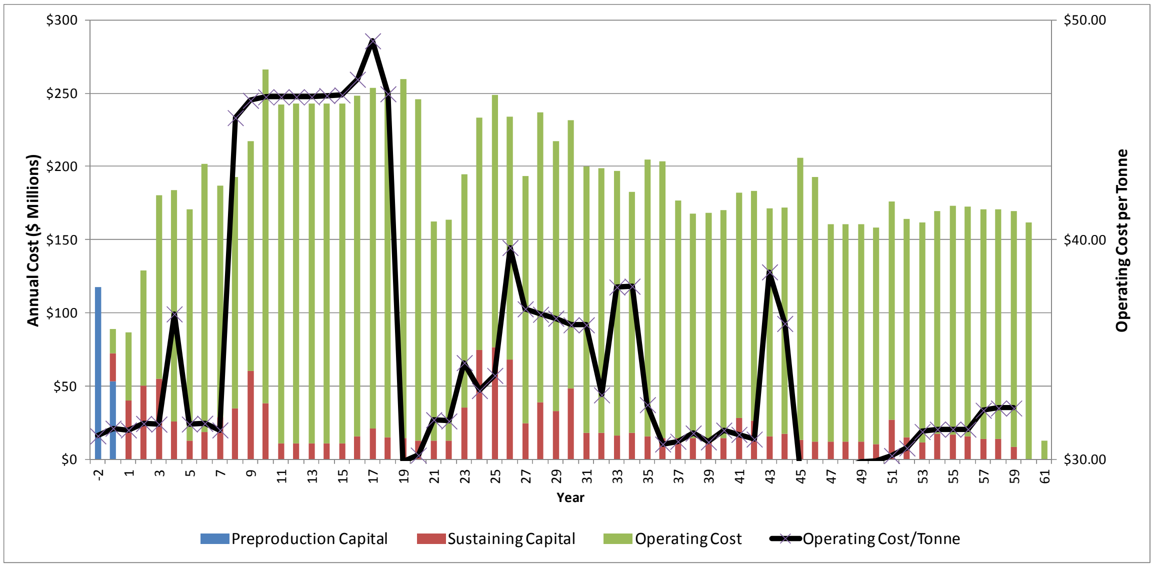
This section describes the parameters, basis, and exclusions for capital estimates for mining the Kamoa Resource. The overall cost estimate for the 5 Mtpa production was compiled to include pre-production capital, sustaining capital, and mine operating costs. The capital costs are estimated at a conceptual study level of accuracy (± 30%). All costs are based on 2011 US$. The unit costs are based on the most recent cost information from similar projects and adjusted where required to fit the mine plan. A summary is provided of pre-production, sustaining capital, and total capital costs for 5 Mtpa and is shown in Table 21-6.

Table 21-6: Underground Cost Summary for 5 Mtpa

Description	Cost
Total Plant Feed Mined Mt	294.9
Preproduction Capital Costs US$M	172.8
Sustaining Capital Costs US$M	1,146.3
Mine Operating Costs US$M	10,365.6
Total Life-of-Mine Expenditures US$M	**11,684.7**
Average Cost US$/t	**39.62**

The annual cost expenditures are summarised in Figure 21-1.

Figure 21-1: Underground Mining Expenditure Schedule – 5 Mtpa



21.4.1 Underground Capital Costs

The total capital cost includes both pre-production capital and sustaining capital. Pre-production capital includes all direct and indirect mine development and construction costs prior to the start of plant feed production. The cost of initial equipment required for use by the contractor for the pre-production development is also included in the pre-production costs. After the initial development is completed by the underground contractors, the equipment fleet used for pre-production will be used for additional sustaining development within the mine.

Sustaining capital is comprised of ongoing capital development and construction as well as mobile equipment rebuild and replacement costs. A capital contingency of 25% is included in the estimate. The pre-production and sustaining capital costs for 5 Mtpa are summarized in Table 21-7. The capital costs included in this evaluation are discussed below.

Table 21-7: Underground Capital Cost Summary – 5 Mtpa

Description	Pre-production US$M	Sustaining US$M	Total US$M
Contractor Costs			
Direct Costs	34.7	–	34.7
Indirect Costs	10.8	–	10.8
Margins/Insurances/Bonding	8.0	–	8.0
Subtotal Contractor Costs	**53.4**	–	**53.4**
Owner Costs			
Surface Capital Infrastructure	3.0	–	3.0
Permanent Fixed Capital Equipment	28.9	60.7	89.6
Permanent Mobile Capital Equipment	19	226.9	245.9
Equipment Rebuild and Replacement	–	373.9	373.9
EPCM	7.8	–	7.8
Owner's Team	3.1	–	3.1
Builder's Risk Insurance	0.3	–	0.3
Power During Pre-production	3.0	–	3.0
Lateral/Vertical Development and Construction	18.3	174.7	193.1
Miscellaneous Construction – Undefined Allowance	–	259.0	259.0
Subtotal Owner's Costs	**83.5**	**1,095**	**1,178.7**
Total Contractor and Owner's Costs	**136.9**	**1.095**	**1,232.1**
Contingency	**34.2**	**232.3**	**266.5**
Total Capital Cost	**171.1**	**1,328**	**1,498.7**

Note: Difference in totals due to rounding.

Contractor Costs

Contractor costs include cost elements listed below.

- Labor
- Permanent Materials
- Equipment – Direct Charge
- Equipment Operating Costs
- Service and Supplies
- Subcontractors.

Descriptions of the key aspects of each cost component are noted below.

Labor

Labor includes a combination of direct and indirect labour required to complete the specified task. Labor can include hourly and staff personnel, depending on the type of activity.

Permanent Materials

Permanent materials includes all materials installed or consumed while performing the specified task, such as concrete, timber, support steel, etc. It is assumed that the contractor provides all permanent materials.

Equipment – Direct Charge

Direct charge equipment includes specialized equipment written off by the Contractor while performing the specified task. Rental rates are not applied to this equipment; it is either entirely written-off or salvaged at work completion. Items that fall into this category include work stages, concrete forms, etc.

Service and Supplies

Service and supplies includes consumable items such as explosives, drilling costs, pipelines, ventilation duct, small tools, etc, associated with the specific task.

Subcontractors

Subcontractors includes subcontractor costs associate with the specific task, such as drain hole drilling, diamond drilling, assaying, etc.

Contractor Indirect Costs

Contractor indirect costs include costs incurred by the contractor to complete specific mine development and construction activities, but are not included in any direct capital cost items.

An assessment will be made of contractor indirect costs during preproduction. The costs are estimated on a daily basis and extended by the activity duration. The Contractor indirect costs include the items listed below.

- Supervision Labor

- Mechanical and Electrical Maintenance Labor
- Temporary Surface Support Equipment Rentals (compressors, generators, fans, wash/dry trailers, office trailer, pickup trucks, etc.).
- Operating Costs for the Temporary Support Equipment
- Service and Supply Costs (sustaining freight, phone/fax, safety and supplies, sewage/garbage disposal, etc.).

Underground Owner Costs

Permanent Capital Equipment

Permanent capital equipment includes the costs associated with purchasing fixed and mobile equipment. In addition, rebuild and replacement costs are assessed against mobile equipment. Data from other recent projects was used to develop permanent capital equipment costs.

To assess permanent capital costs, equipment lists are developed from infrastructure designs and operating parameters. Once an equipment list is compiled, and then purchase, rebuild, and replacement costs are estimated. Following are the key elements used to develop the unit cost database.

- Item Description – identifies and sometimes provides a brief technical description of the equipment duty requirements or capacity.
- Base Cost – a base cost as quoted by a vendor or taken from a historical cost database, including the cost for options.
- Development Allowance – a 5% allowance to cover the cost of miscellaneous components, fuels, lubricants, and services required to commission a piece of equipment.
- Spares Allowance – a cost allowance for spare parts required on-site. When provided, the cost of spares recommended by the vendor is included. Similarly, engineering judgment is used to reduce or eliminate spares allowances for identical units that appear in multiple equipment list areas. The following guidelines are used to calculate spare parts costs.
  - Single units – 5%
  - Pairs of units – 5%
  - Multiple units – 5%

- Freight Allowance – a cost allowance for delivering equipment to site. An average of 5% of the base cost is used when a vendor quotation is not provided.
- Total Unit Cost – a summation of the base cost, development allowance, spares allowance, and freight allowance all escalated into current terms, if necessary. The total cost excludes sales tax and contingency.

Once base unit costs are developed, mobile equipment rebuild and replacement unit costs are estimated. In general, these costs are based on annual operating hours, and estimates of the average life to rebuild and replacement, which varies to suit the type of equipment. For the purposes of this evaluation, an annual allowance was estimated at 15.0% of new equipment value for all mobile equipment.

Engineering, Procurement, and Construction Management

Engineering, procurement, and construction management (EPCM) costs are determined as a 7.5% allowance assessed against the "Total Contractors Cost + Surface Infrastructure Cost + Permanent Fixed Capital Equipment Costs + Permanent Mobile Capital Equipment Costs."

Owner's Team

The Underground Owner's Team cost may include a manager, mine clerk, and mine engineer. Costs include labour, surface pickup operating costs, permanent equipment costs, and miscellaneous costs to support the Owner's team from the start of the preproduction to the start of production. The cost represents an assessment of a 3% allowance against the "Total Contractors Cost + Surface Infrastructure Cost + Permanent Fixed Capital Equipment Costs + Permanent Mobile Capital Equipment Costs."

Builder's Risk Insurance

Builder's risk insurance is assessed at 0.3% against the "Total Contractors Cost + Surface Infrastructure Cost + Permanent Fixed Capital Equipment Costs + Permanent Mobile Capital Equipment Costs."

Electric Power Consumption

Power costs associated with pre-production mine development and construction are assessed at a calculated rate of US$0.075 per kWh. This will need to be updated with actual costs when available.

Contingency

A capital contingency of 25% is assessed against the total Contractors and Owners costs. The contingency provides additional project capital for expenditures that are anticipated, but not defined, due to the level engineering detail in the study.

Capital Cost Criteria and Assumptions

Capital mine development work performed during preproduction is performed by a contractor. After production begins, Owner's personnel perform the work.

The estimates are based on the following qualifications.

Pre-production Costs

- Costs are based on constant 2011 US Dollars.
- Pre-production contractor crews will work seven days per week, two shifts per day, and 11 hours per shift.
- Five days per year are allowed as non-production days.
- A contractor provides all underground equipment during the pre-production period except mobile equipment, which will be purchased by the owner and operated by the contractor during the preproduction period.
- Contractor's margins/insurances are assessed at 17.5%.

Quantity Development

The design engineering team developed all quantities included in, associated with, or related to the scope of work based on conceptual mine designs.

A mine equipment list was developed based on the infrastructure design and production productivity. A major portion of the equipment prices is based on industry averages and historical data from similar projects.

Exclusions

The following services and facilities are not included in the capital cost estimates.

- Escalation
- Finance Charges and Interest Charges
- Land Acquisitions, Rights-of-Way, and Licenses
- Disposal of Hazardous Materials (if any).

21.4.2 Underground Mine Operating Costs

Unit operating costs were prepared for Room-and-Pillar stoping methods and for Drift-and-Fill stoping methods with CoRF, and annual operating costs were generated based on the tonnes produced in each year. A summary of the operating costs (including pre-production costs) can be found in Table 21-8.

Table 21-8: Underground Operating Cost Summary

Description	Total Cost (Millions US$)	Cost per Tonne (US$)
Production Direct Costs	6,902.8	23.41
Indirect Operating Costs	2,070.9	7.02
Power Costs	449.8	1.53
Undefined Allowance	942.3	3.20
Total Operating Cost	**10,365.8**	**35.15**

Operating Cost Estimate Scope Definition

These cost estimates are intended to cover all expenses required to operate the mine and produce plant feed. The operating costs are subdivided into the following cost centers.

Development (Lateral and Vertical)

Development includes all resource development (drill/draw drift development and drill drift slashing), remaining lateral waste development (ramp, footwall, stope access, and miscellaneous excavation), drop raising, exhaust raises and orepasses, and grizzly construction will be performed by the Owner’s personnel.

Production Direct Costs

Production includes the costs to drill, blast, ground support, and muck the minerals from the stopes into the dedicated haulage trucks or orepasses and mineral handling to surface.

CoRF Costs

CoRF costs consists of all costs associated with preparing and filling stopes including cement costs and freight, backfill plant operating costs, and operating labour, and delivery costs.

Indirect Operating Costs

Includes the cost of pumping, ventilation, providing compressed air, mine service crews and trainees/spares, mechanics and electricians for mine operations, and site staff personnel for the mine.

Power Costs

Representative power costs were based on power loads developed from comparable projects of similar size. A detailed breakdown for power was not prepared at this level of evaluation.

Miscellaneous Construction – Undefined Allowance

An allowance of 10% of operating costs has been included for miscellaneous underground excavations, which may be required for efficient operation of the Project.

Operating Cost Exclusions

In addition to capital construction related costs, the following operating cost items are excluded from the compiled operating costs.

- Federal or local sales taxes on permanent materials or services.
- Milling, refining, or shipping costs.
- Contingency on operating costs.
- Finance charges and interest charges.
- Land acquisition, rights-of-way, licenses, and royalties.
- Disposal of hazardous materials.
- Surface plant feed and waste haulage.

21.4.3 Underground Personnel Requirements

Personnel requirements were estimated using the cost model approach. Requirements for each operating sector were assessed and an annual personnel estimate was prepared. Stantec assumes that expatriate staff and management will be required in the initial stages of the project and will train the Congolese staff during the first 10 years of the project. Additional expatriate staff may be required during the life-of-mine to oversee the mining project and will need to be defined based on the local regulations. Table 21-9 shows the personnel requirement at full production.

Table 21-9: Personnel Requirement at Full Production

Expatriate	
Managers	2
Superintendents	3
Foremen	13
Engineers	11
Geologists	9
Shift Bosses	17
Technicians	13
Total Expatriate Personnel	**68**
Hourly	
Electricians	78
Mechanics	162
Maintenance Workers	86
Helpers	129
Underground Labourers	108
Surface Labourers	43
Total Hourly Personnel	**606**
Salaried	
Managers	5
Superintendents	16
Foremen	38
Engineers	19
Geologists	22
Shift Bosses	60
Technicians	38
Accountants	19
Purchasing	30
Personnel	38
Secretaries	52
Clerks	66
Total Salaried Personnel	**403**
Total	**1,077**

21.5 Process Plant Operating Costs

The results are presented as those for the concentrator, those for the smelter and a summary total for the entire surface plants, related back to a common run-of-mine (ROM) input. The estimates include the process consumables, maintenance & spares, labour and power. The estimates exclude mining, security, environmental monitoring, head/corporate office costs, transport, insurance, tailings dam operation and site closure and rehabilitation.

It is estimated from similar operations that the staffing requirements for the concentrator are 209 and those for the Smelter are 166 giving a total staff complement of 375 personnel. Note that this figure excludes general administrative staff but includes maintenance personnel.

The unit costs are based on an annual production of 5,000,000 t and 365 days/annum operation at 92% availability. The unit of currency is the US dollar (US$). The cost basis has been brought to 2011 and the accuracy is in the range +30% to -25%.

The estimated long term process operating cost (concentrator plus smelter) is approximately US$16.335/t. This does not include power costs.

A summary of the plant operations for a typical year is given in Table 21-10. The table does not include the power costs that have been incorporated in the financial analysis in Section 22. All the costs are exclusive of import duties, withholding tax, taxes or other government required costs unless otherwise noted. The following aspects have been included in the estimate:

- This estimate is based on pricing data current to second quarter of 2011 and has ±30% accuracy.
- Workforce requirement includes supervision, operation and maintenance. The rates used are from a recent AMEC study for a project in the DRC. The rates include allowances for housing and services.
- Water costs for fresh water make-up for the concentrator and smelter are based on the pumping power required to transport the water over a distance of 25 km and at an elevation of +100 m, with an estimate included for maintenance and treatment chemicals. This excludes any licence fees, abstraction fees, etc.
- Maintenance costs have been factored from capital costs scaled according to throughput.
- Reagent costs have been based on test work and budget prices.

Table 21-10: Summary of Plant Operations

– Description	– Labour	– Annual Cost - US$M	– Unit Cost - US$/t ROM
– **Concentrator**			
– **Workforce**			
– **Workforce**	– **209**	– **6.30**	– **1.259**
– **Major Consumables**			
– Reagents	–	– 3.98	– 0.795
– Laboratory	–	– 1.49	– 0.298
– Mill Balls	–	– 23.18	– 4.635
– Water	–	– 0.54	– 0.107
– Others	–	– 1.68	– 0.336
– **Subtotal Consumables**	–	– **30.85**	– **6.171**
– **Supplies**			
– Maintenance	–	– 2.41	– 0.481
– Operating	–	– 0.83	– 0.166
– Power	–	– –	– –
– **Subtotal Supplies**	–	– **3.23**	– **0.647**
– **Total - Concentrator Process**	–	– **40.39**	– **8.077**
– **Smelter**			
– **Workforce**			
– **Subtotal Workforce**	– **166**	– **8.44**	– **1.689**
– **Major Consumables**			
– Reagents & Consumables	–	– 27.01	– 5.402
– Laboratory	–	– 0.12	– 0.024
– Cooling Water	–	– 0.18	– 0.036
– Others	–	– 1.40	– 0.280
– **Subtotal Consumables**	–	– **28.68**	– **5.736**
– **Supplies**			
– Maintenance	–	– 4.16	– 0.833
– Operating	–	– –	– –
– Power	–	– –	– –
– **Subtotal Supplies**	–	– **4.16**	– **0.833**
– **Total - Smelter Process**	–	– **41.29**	– **8.258**
– **Total -** Workforce	– 375	– 14.74	– 2.948
– **Total -** Consumables	–	– 59.53	– 11.906
– **Total -** Supplies	–	– 7.40	– 1.480
– **Total - Process**	– **375**	– **81.67**	– **16.335**

The estimated total workforce unit cost is US$2.95/t ROM. A total of 375 personnel are required for the total process operation of concentrator and smelter. This includes 27 management and professional services staff, 208 operators for operations and laboratory and 104 personnel for maintenance. The estimate assumes 8 hr/shift, 24 hr/day and 365 days/annum.

Major metal consumables are estimated at US$4.72/t ROM. These include mill balls, mill and crusher liners.

Other consumables include office stationary and equipment, IT supplies and maintenance tools as well as ancillary social and welfare services.

Maintenance supplies are factored from the capital cost. The total operating and maintenance unit cost is US$1.48/t ROM.

The power cost is derived to be US$2.52/t ROM using the installed power of 55 MW and a unit price of USc3.5/kWh. This includes the oxygen plant for the smelter.

21.6 Process Plant and Infrastructure Capital Costs

AMEC prepared costs for process plant and infrastructure areas. The process plant and infrastructure includes: the concentrator, smelter, tailings storage facility (TSF) and infrastructure, including power supply. The estimated amounts cover the direct and indirect costs for all equipment, temporary facilities, materials and labour required to construct and complete the permanent works. Direct costs are those expenditures that include supply of capital equipment and materials, freight costs to the construction site and construction labour for the installation. Indirect costs are those expenditures covering engineering, procurement and construction management (EPCM) services, together with the supervision of commissioning of the works. An allowance for development has been made as well as sustaining capital over the 60 year life-of-mine (including pre-production). The capital cost is estimated at a Scoping Study level of accuracy. Costs are as summarized in Table 21-11.

Table 21-11: Summary of Process and Infrastructure Capital Costs

US$M	Phase 1	Sustaining	Total
Power & Smelter			
Smelter	313	38	351
Power	141	85	226
Subtotal	**455**	**122**	**577**
Concentrator & Tailings			
Concentrator	270	12	281
Tailings	42	372	414
Subtotal	**312**	**383**	**696**
Infrastructure			
Site Facilities and Temporary Works	234	141	375
Off-site Infrastructure	24	14	38
Subtotal	**257**	**155**	**413**
Indirects			
Construction Indirects EPCM	155	2	157
Subtotal	**155**	**2**	**157**
Owners Cost (incl. Drilling & Studies)			
Owners Cost	205	92	298
Closure		200	200
Subtotal	**205**	**292**	**498**
Capital Expenditure Before Contingency	**1,385**	**955**	**2,341**
Contingency	368	453	821
Capital Expenditure After Contingency	**1,753**	**1,409**	**3,162**

The process and infrastructure capital cost includes both pre-production capital and sustaining capital. The process and infrastructure contingency applied is 25%, except for power, which is 35% and which is in line with the level of study, engineering progress and preliminary nature of plant design and engineering. The process and infrastructure sustaining capital is calculated as shown in Table 21-12.

Table 21-12: Process and Infrastructure Capital

Annual Sustaining Capital	%	% of
Concentrator	0.2	Equipment cost
Smelter	0.2	Total cost
Off-site Infrastructure	1	–
Tailings	0.1	Total Cost
Site Facilities and Temporary Works	1	Total cost
Off-Site Infrastructure	1	Total cost
Bore Field	1	Total cost
Power Plant	1	Total cost

The estimating approach for the process plant was to price all major process/mechanical equipment. The price includes delivery to site and any vendor recommended installation and commissioning spares: note that operating and maintenance spares are included in the operating expenditure. The equipment supply costs were then used as the basis for estimating costs for the balance of codes of accounts (civil/concrete, piping, structural steel, electrical and instrumentation).

The process and infrastructure estimate was compiled in South African Rand (ZAR) and converted to USD at a rate of ZAR8.40 to the USD.

The method of project execution applied to the estimate calculation assumes an EPCM contract.

Several options pertaining to the TSF were developed as to select the most suitable method of construction and location of the facility. The outcome of the assessment concluded that TSF Option 1 represented the most favourable for the Scoping Study.

Concentrator

The estimated costs for process plant (excluding the smelter) mechanical equipment are as listed in the mechanical equipment schedules.

These costs have been derived by the following methods.

Short-form enquiries were prepared and issued for all major equipment. This category represented approximately ~95% of the total process plant mechanical equipment cost and included the following:

- Crushers
- Conveyors

- Ball mills
- Cyclones
- Flotation Cells and Blowers
- Feeders and screens
- Concentrate filters
- Thickeners
- Pumps.

Platework costs have been factored from the mechanical equipment costs.

Smelter

The smelter process plant mechanical equipment cost was provided as part of a vendor package.

Various exclusions from this package are listed below:

- Acid storage tanks
- Flux and coal plants
- LFO storage facility
- Water treatment plant
- Concentrate handling facility from concentrate shed to smelter feed
- Stack.

The cost of the items above were derived from the AMEC database or calculated from first principles.

Earthworks

The earthworks cost for the process plant and the smelter were factored from the mechanical equipment costing schedule for each area respectively. The scope of work covers the following:

- Bulk earthworks and plant terracing (incl crushing area)
- Clearing & grub for tailings line

- Raw water dam and lining thereof
- Lined run-off water dam with silt trap
- Maintenance access roads
- Sewage treatment
- Security fencing
- Non-process or architectural plant buildings
- The plant access road (25 km) was included in the estimate.

Civil Works

All civil costs (preparation of foundations, concrete materials and associated labour) have been factored from the mechanical equipment costs.

In the absence of a detailed geotechnical report at this stage, assumptions were made and factored accordingly. The cost for dams (raw water and pollution control dam) were included in the estimate.

Structural Steelwork (Equipment Supports)

Structural steelwork costs have been factored from the mechanical equipment costs for the process plant (excluding the smelter). Structural steelwork associated with conveyor systems (gantries, trestles, take up steelwork, walkways, skirts, etc.) and the smelter is included in the respective package price.

EPCM Costs

These costs cover the project management, engineering (design), procurement, construction management, and commissioning management costs directly associated with the implementation of the project.

Escalation

Escalation cost has been excluded from this estimate.

21.7 General and Administration Costs

General and administration costs were estimated based on labour number and other costs requirements for the administration, information technology, transportation, finance, commercial, procurement, warehousing, legal, environmental, community and government relations, human resources, safety, training, security, accommodation and messing, and further studies. The messing and accommodation costs include the costs of all project personnel in the mining, processing, and general and administration departments. Average general and administration costs are US$5.40/t.

21.8 Owners Costs

Ivanplats have prepared a budget for Owners costs from 2012 through to 2017. The costs include drilling, sampling and assays, further studies for project development, environmental and community, administration. The Ivanplats budget is shown in Table 21-13. In addition to the Ivanplats owners cost budget an allowance of 3% of capital costs has been added to the project capital cost to allow for the management of design and construction.

Table 21-13:Ivanplats Owners Cost Budget

Item	2012 US$M	2013 US$M	2014 US$M	2015 US$M	2016 US$M	2017 US$M	Total US$M
Drilling	15.00	5.00	–	–	–	–	20.00
Sampling and Assaying	0.50	9.30	–	–	–	–	9.80
Studies	1.20	0.80	0.80	0.40	0.16	–	3.36
Salaries (DRC)	1.50	1.00	1.00	0.50	0.20	–	4.20
Administration	0.63	0.25	0.13	0.03	–	–	1.03
Social Obligations	–	5.00	5.00	5.00	–	–	15.00
Capital Expenditure	1.00	1.00	1.00	–	–	–	3.00
Total	**19.83**	**22.35**	**7.93**	**5.93**	**0.36**	–	**56.39**

22.0 ECONOMIC ANALYSIS

22.1 Introduction

This section incorporates assumptions, analysis and estimates related to the 2012 PEA.

The work prepared for Section 22 Economic Analysis of the 2012 PEA has not been updated. It is considered that this work remains current as there has not been any additional work or changes to the assumptions that are likely to result in a material change to the conclusions of Section 22 Economic Analysis of the 2012 PEA.

The QP for Section 22 of this Report is Bernard Peters.

22.2 Summary of Financial Results

The 2012 PEA includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorized as Mineral Reserves, and there is no certainty that the results will be realized. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability. The results of the 2012 PEA represent forward-looking information. The forward-looking information includes metal price assumptions, cash flow forecasts, projected capital and operating costs, metal recoveries, mine life and production rates, and other assumptions used in the 2012 PEA. Readers are cautioned that actual results may vary from those presented. The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially are presented in the body of this report under each relevant section.

The plan described in the study is for the construction and operation of a long-term underground mine, concentrator processing facility and smelter operation and associated infrastructure. The mining rate and concentrator feed capacity is 5 Mtpa, producing 143 ktpa of blister copper on average in the first 10 years of operation. The production scenario schedules a mineral inventory (which includes indicated and inferred resources) of 299 Mt over 61 years, producing 7.8 Mt of blister copper.

The economic analysis used a long term price assumption of US$2.85/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 a Net Present Value (NPV) at a 10% discount rate of US$1.16 billion (after tax). It has an after tax internal rate of return (IRR) of 17.0% and a payback period of 5.29 years. The average total cash cost after credits in the first 10 years of production is US$0.95/lb of copper. Table 22-2 summarises the financial results, whilst Table 22-3 summarises mine production, processing and concentrate, and metal production statistics. Realisation costs are shown which indicate the actual realizable value of Payable Copper produced after accounting for the transport treatment and DRC Government royalties payable on these sales.

22.2.1 Democratic Republic of Congo Fiscal Environment

A Mining Code (Law No. 007/2002 of July 11, 2002) (2002 Mining Code) governs prospecting, exploration, exploitation, processing, transportation, and the sales of mineral substances.

22.2.2 Model Assumption

Pricing and Discount Rate Assumptions

The Project level valuation model begins on 1 January 2012. It is presented in 2012 constant dollars, cash flows are assumed to occur evenly during each year and a mid-year discounting approach is taken. The discount factor applied to the base case Project analysis is 10%.

The copper price used for the evaluation is US$2.85/lb copper. An acid price of US$250/t has also been used in the financial model. These prices were considered to be reasonable based on industry forecasts and prices used in other studies.

The product being sold is blister copper and payment terms for the copper assume that the payable copper is 99.7% of the blister copper content with a blister refining charge of US$100/t. The blister transport charge to the customer is assumed to be US$380/t.

22.2.3 Taxation

The DRC Mining Code provides for all the taxes, charges, royalties, and other fees. Ivanplats engaged KPMG Canada, 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 basedare calculated in on nominal dollars and outputs are then deflated for use in the integrated cash flow calculation. The working capital assumptions for receivables, payables and inventories are 6 weeks, 4 weeks and 3 months respectively. These assumptions are preliminary and will need to be verified in later studies.

Corporate Taxation

Corporate Tax will be applied at a rate of 30% of taxable income. Taxable income will be calculated on a simple taxable profit basis calculated without consideration of specific deductions or allowances that may or may not exist in the legislation.

The DRC has a taxation system where business profits sourced by companies in the DRC are subject to income tax and distributions of net profits are subject to tax on income from movable capital. Companies that are the holders of mining rights are subject to tax at 30% (otherwise 40%) on net income (calculated by deducting all acceptable expenditure and depreciation of fixed assets from gross income derived during a year of assessment) and withholding tax on distributions are subject to 10% tax (otherwise 20%) at the shareholder's level.

It is assumed that no special deductions or allowances will be taken into account in the calculation of taxable income for purposes of the financial model (i.e. net income will be calculated on a simple taxable profit basis).

Although it has been reported that a tax holiday incentive may be recognized allowing for a 3–5 year period where a 0% tax rate will apply, this requires further clarification. Consequently, it has been assumed in the financial model that no tax holiday is applicable, because the DRC tax legislation does not currently provide for any tax holiday incentives.

An initial assumption is made that a 5 year tax loss carry forward period is applicable.

In the DRC tax losses can be distinguished between losses incurred during exploration, and losses incurred once mining production commences.

Since no income is earned during the exploration period, exploration expenditure incurred may only be deducted once production begins. The aggregate exploration expenditure may be claimed as an equal deduction over a two year period once production commences. To the extent that the deduction of exploration expenditure creates an assessed loss, this loss should be ring-fenced and may be utilised in subsequent years without any limitation.

Companies in an assessed loss position arising from operational activities may carry forward such assessed losses for 5 years upon receipt of prior approval from the tax authorities.

Royalties

According to the 2002 Mining Code 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 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. Different rates apply to different types of metals sold. The holder of the Exploitation Permit will benefit from a tax credit equal to a third of the mining royalties paid on products sold to a transformation entity located in the DRC. Mining royalties paid may be deducted for income tax purposes.

Depreciation

Non-mining assets are depreciated in accordance with the common law. Specific mining assets dedicated to mining operations, with useful lives of between 4 and 20 years are depreciated as follows:

- First year: 60% depreciation based on the cost of the asset; and
- For subsequent years: a declining balance depreciation method is applied based on the tax years remaining over the life of the mine.

It should be noted that depreciable items which are normally utilised for a period of less than 4 years or a period of more than 20 years will not qualify to use the declining balance method and will fall under the common law provisions. The common law provides different depreciation rates for various assets (e.g. 10 years for furniture and 5 years for vehicles).

Depreciation arising in loss yielding tax periods is considered to be deferred and may only be set off against taxable income in future years. The deferral is not subject to any time limitation. The asset useful lives used in the financial model are shown in Table 22-1.

Table 22-1: Asset Useful Life

Asset Type	Useful Life (years)
Building and construction	20 years
Concentrator, Power, Smelter & Mining Capital	3 years
Machinery and equipment - Other	10 years
Furniture, Fitting and Fixtures	10 years
Motor Vehicles	4 years

VAT

VAT will be included as a bottom line calculation for 75% of all operating costs and 100% of capital costs at a rate of 16%. This VAT; however, will be assumed to be fully refunded in the year following the year of imposition so that VAT becomes a working capital adjustment only.

VAT came into effect in the DRC in January 2012. Previously, a turnover tax at a reduced rate of 5% was applied to import transactions and the sale of locally made products and services rendered. 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 and operating costs for direct cost line items on the following basis:

- 2% Pre-Production capital
- 5% Post-Production capital
- 3% Operations (75% of all operating costs).

In order to qualify for favourable import duty rates, a mining company before commencing work, is required to submit a list detailing the number and value of all moveable property, equipment, vehicles, mineral substances and other items to be used at all stages of the mining process, for approval to the Ministries of Mines and Finance. If no reply is received within 30 working days following the date of the receipt of the application letter, the list is deemed to have been approved, and the relevant authorities must issue the Decree of approval, within 7 days. Once approval is obtained, a favourable import duty rate of 2% will apply to items on the list.

The rate of 2% only applies to the period before the effective commencement of exploitation work.

A rate of 5% will apply for items on the abovementioned list upon the effective commencement of exploitation work. It should be noted that the rate of import tax is based on whether exploitation activities have commenced and not whether production has commenced.

The supplies of consumer goods, reagents and maintenance products needed for everyday use, but not directly related to mining activities, are excluded from the said lists.

Imports of equipment, goods, tools, and other items which do not appear on the approved lists, are subject to the provisions of the substantive law.

Fuel, lubricants, reagents, and consumer goods destined for mining activities are subject to a single import duty of 3% throughout the duration of the project.

Export Taxes

A 1% export tax will be assumed on an NSR basis for all revenues. This fee is more broadly regarded as a government levy for certain procedures (e.g. quality control) that must be conducted in respect of exports. The fee is however limited to 1% of the value of the export.

Excise Tax

No excise taxes will be assumed in the models, all such taxes being considered in the generation of cost inputs. No excise taxes should apply in the current instance.

Withholding Taxes on Services

No withholding taxes on services will be assumed in the economic assessment, all such taxes being considered in the generation of cost inputs.

The withholding tax referred to above has been abolished since VAT has come into effect in January 2012. Foreign companies are now subject to 16% VAT on services rendered in the DRC, and must elect a tax representative in the DRC for administration purposes in order to account for such VAT.

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%.

Exceptional Tax on Expatriates

In the DRC an employer is liable for the exceptional tax on expatriate's remuneration at a rate of 10%. It is determined in terms of the salaries generated by the work carried out in the DRC, and is not deductible for purposes of calculating the income tax payable. It has been assumed that this tax is included in the unit rates.

22.2.4 Case Overview and Results

This report includes the results of the 2012 PEA and includes an economic analysis that is based, in part, on Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would allow them to be categorized as Mineral Reserves, and there is no certainty that the results will be realized. Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability. The results of the 2012 PEA represent forward-looking information. The forward-looking information includes metal price assumptions, cash flow forecasts, projected capital and operating costs, metal recoveries, mine life and production rates, and other assumptions used in the 2012 PEA. Readers are cautioned that actual results may vary from those presented. The factors and assumptions used to develop the forward-looking information, and the risks that could cause the actual results to differ materially are presented in the body of this report under each relevant section. The 2012 PEA analyses a single production scenario. The work; however, has indicated that other scenarios are likely to show improved results. AMC has recommended that further studies are undertaken to assess the alternative production scenarios.

The plan described in the 2012 PEA is for the construction and operation of a long-term underground mine, concentrator processing facility and smelter operation and associated infrastructure. The mining rate and concentrator feed capacity is 5 Mtpa, producing 143 ktpa of payable copper on average in the first 10 years of operation. The production scenario schedules 299 Mt over 61 years, producing 7.8 Mt of blister copper. The production schedule includes indicated and inferred resources.

The economic analysis used a long term price assumption of US$2.85/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 a Net Present Value (NPV) at a 10% discount rate of

US$1.16 billion (after tax). It has an after tax internal rate of return (IRR) of 17.0% and a payback period of 5.29 years. The average total cash cost after credits in the first 10 years of production is US$0.95/lb of copper. Table 22-2 summarises the financial results, whilst Table 22-3 summarises mine production, processing and concentrate, and metal production statistics.

Table 22-2: Financial Results

		Before Taxation	After Taxation
Net Present Value US$M	Undiscounted	25,430	17,640
	5.0%	6,121	4,062
	8.0%	3,091	1,929
	10.0%	2,002	1,164
	12.0%	1,286	663
	15.0%	612	196
IRR	–	20.5%	17.0%
Project Payback (years)	–	4.63	5.29

Table 22-3: Mining and Processing Production Statistics

	Total LOM	First 5 Years Average	First 10 Years Average	LOM Average
Total Plant Feed Mined ('000 t)	299,391	4,463	4,731	4,908
Quantity Plant Feed Treated ('000 t)	299,388	4,800	4,900	4,908
Copper Feed Grade (%)	3.14	3.29	3.50	–
Copper Recovery (%)	84.8%	84.3%	85.3%	–
Concentrate Produced ('000 t)	24,384	411	450	400
Copper Concentrate Grade (%)	32.7%	32.3%	32.6%	–
Contained Metal in Concentrate				
Copper ('000t)	7,984	133	147	131
Copper (Mlb)	17,601	293	324	289
Payable Copper				
Copper ('000t)	7,801	130	143	128
Copper (Mlb)	17,198	287	316	282

Note: First 5 Years Average and First 10 Years Average plant feed mined does not include mined material prior to Year 1 (1,688 kt)..\

Figure 22-1 and Figure 22-2 depict the processing, concentrate and metal production, respectively.

Figure 22-1: Plant Feed Processing

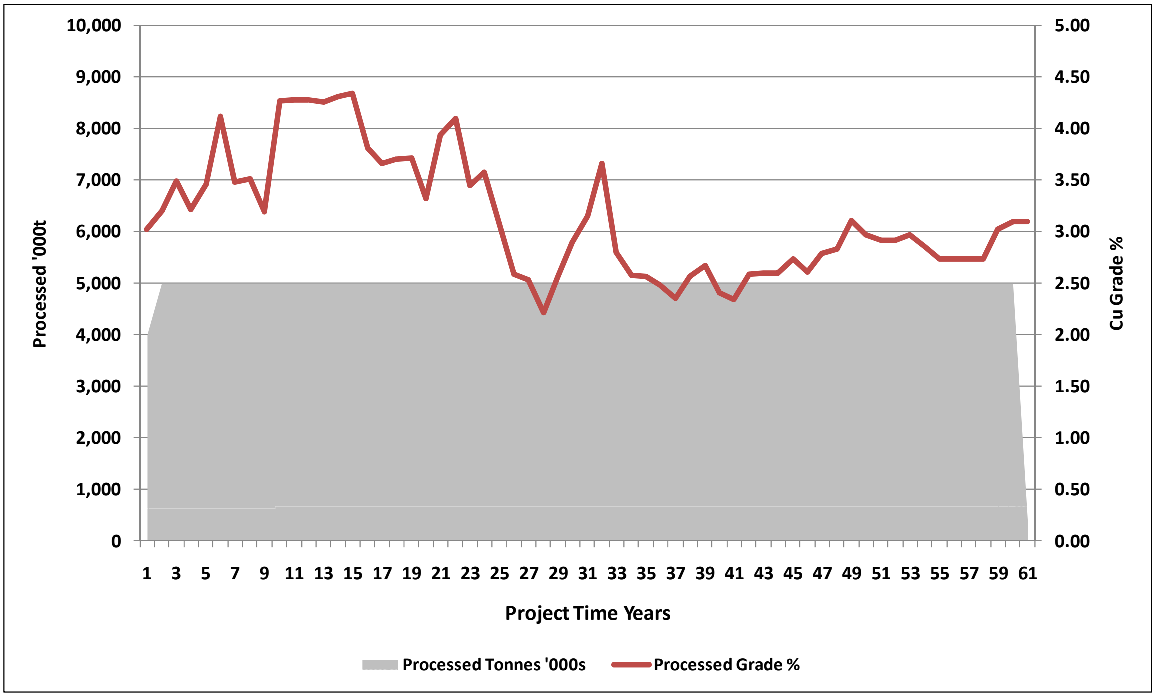


Figure 22-2: Concentrate and Metal Production

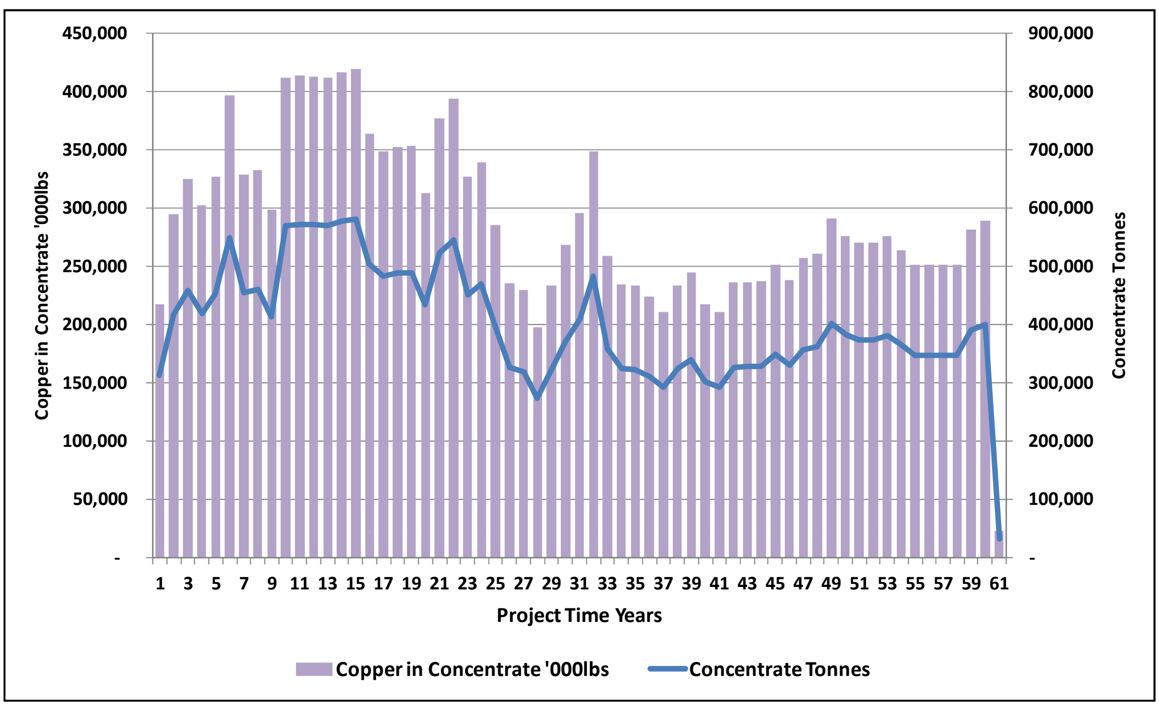


Table 22-4 summarises unit operating costs and Table 22-5 provides a breakdown of operating costs and revenue.

Table 22-4: Unit Operating Costs

	US$/lb Payable Copper		
	LOM Average	**First 5 Years Average**	**First 10 Years Average**
Mine Site Cash Cost	1.04	0.93	0.89
Realisation Cost	0.30	0.30	0.30
Total Cash Costs Before Credits	**1.34**	**1.22**	**1.19**
Acid Credits	0.24	0.24	0.24
Total Cash Costs After Credits	**1.10**	**0.99**	**0.95**

Table 22-5: Operating Costs and Revenues

	US$M	**US$/t Plant Feed Milled**		
	Total LOM	**First 5 Years**	**First 10 Years**	**LOM Average**
Revenue				
Copper in Blister	49,013	170.16	183.85	163.71
Acid	4,047	14.23	15.26	13.52
Gross Sales Revenue	**53,061**	**184.39**	**199.11**	**177.23**
Less: Realisation Costs				
Transport	2,988	10.37	11.21	9.98
Treatment and Refining	786	2.73	2.95	2.63
Royalties and Export Tax	1,357	4.71	5.09	4.53
Total Realisation Costs	**5,132**	**17.82**	**19.25**	**17.14**
Net Sales Revenue	**47,929**	**166.57**	**179.86**	**160.09**
Site Operating Costs				
OP & UG Mining	10,476	28.92	32.20	34.99
Processing	3,125	10.55	10.45	10.44
Smelting	2,203	7.65	7.90	7.36
Tailings	83	0.27	0.27	0.28
General & Administration	1,616	6.87	5.63	5.40
Customs	342	1.01	1.09	1.14
Total	**17,844**	**55.28**	**57.54**	**59.60**
Operating Margin	**30,085**	**111.29**	**122.32**	**100.49**

The capital costs for the project are detailed in Table 22-6.

Table 22-6: Capital Investment Summary

US$M	Phase 1	Sustaining	Total
Mining			
Open Pit Mining	63	5	67
Underground Mining	173	1,146	1,319
Subtotal	**235**	**1,151**	**1,386**
Power & Smelter			
Smelter	313	38	351
Power	141	85	226
Subtotal	**455**	**122**	**577**
Concentrate & Tailings			
Concentrator	270	12	281
Tailings	42	372	414
Subtotal	**312**	**383**	**696**
Infrastructure			
Site Facilities and Temporary Works	234	141	375
Off-site Infrastructure	24	14	38
Subtotal	**257**	**155**	**413**
Indirects			
Construction Indirects EPCM	155	2	157
Subtotal	**155**	**2**	**157**
Owners Cost (incl. Drilling & Studies)			
Owners Cost	205	92	298
Closure	–	200	200
Subtotal	**205**	**292**	**498**
Capital Expenditure Before Contingency	**1,621**	**2,106**	**3,727**
Contingency	368	453	821
Capital Expenditure After Contingency	**1,989**	**2,560**	**4,549**

The cash flow sensitivity to metal price variation is shown in Table 22-7, for copper prices from US$2.00/lb Cu to US$4.00/lb. The Project cash flow includes revenue from acid that would be produced in the smelter. The credit from acid revenue represents 8% of gross revenue and Table 22-8 indicates this is approximately 20% of the After Tax NPV_{10} at a copper price of US$2.85/lb Cu. If an acid price of US$400/t were achieved from sales then the After Tax NPV_{10} would be increased by 18%.

The sensitivity of After Tax NPV_{10} to initial capital cost, direct operating costs, transport and Cu feed grade is shown in Table 22-9. The table shows the change in the base case After Tax NPV_{10} of US$1,989 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.

Table 22-7: Metal Price Sensitivity

	Copper Price - US$/lb				
Net Present Value	**2.00**	**2.50**	**2.85**	**3.50**	**4.00**
5.0%	1,333	2,941	4,062	6,141	7,739
8.0%	322	1,271	1,929	3,149	4,086
10.0%	-34	674	1,164	2,071	2,766
12.0%	-260	287	663	1,360	1,893
15.0%	-457	-70	196	686	1,061
IRR	9.8%	14.3%	17.0%	21.5%	24.6%

Table 22-8: Acid Price Sensitivity

Acid Price US$/t	–	100	150	250	400	600
NPV_{10} US$M	823	959	1,027	1,164	1,368	1,641
% Change	-29%	-18%	-12%	0%	18%	41%

Table 22-9: Additional Sensitivities

			Change from Base NPV_{10} US$ M		
Variable	**Units**	**Base**	**-10.00%**	**+10.00%**	**+25.00%**
Initial Capital	**US$ M**	**1,989**	141	(141)	(353)
Direct Operating Costs	**US$/t**	**35**	265	(291)	(778)
Transport Costs	**US$/t**	**380**	25	(25)	(62)
Cu Feed Grade	**% Cu**	**3.14**	(402)	402	988

Cumulative cash flow is depicted in Figure 22-3. The Project cash flow is shown in Table 22-10.

Figure 22-3: Cumulative Cash Flow

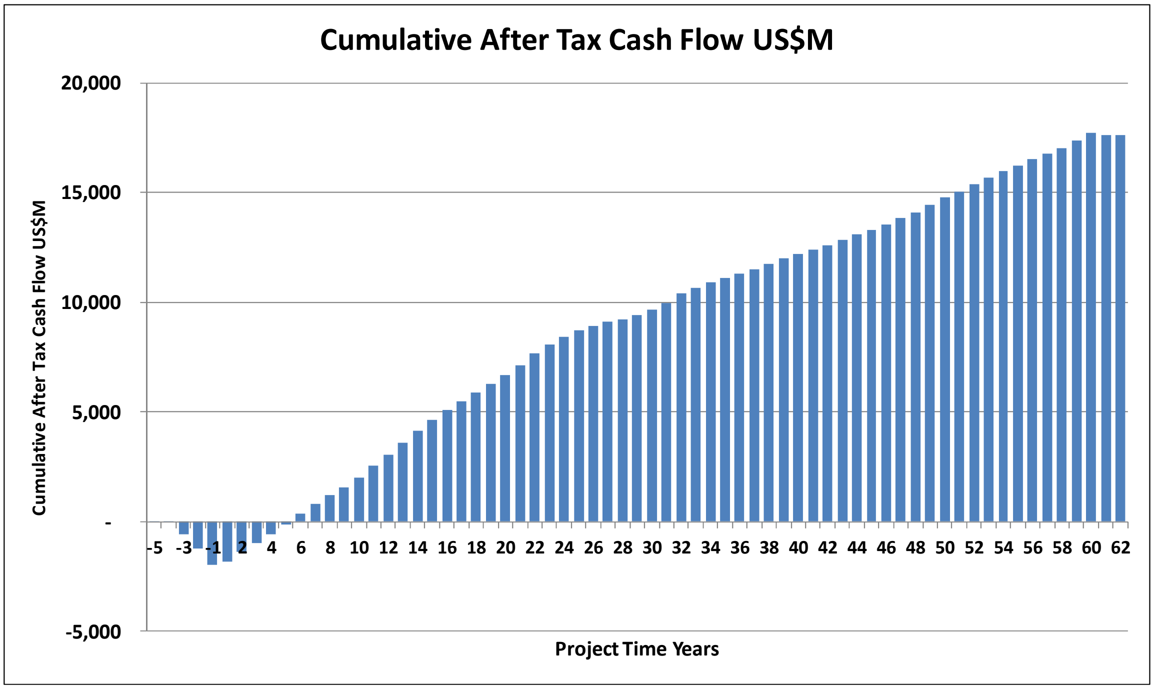


Table 22-10: Cash Flow

Year	-5	-4	-3	-2	-1	1	2	3	4	5	6 to 10	11 to 20	20 to LOM	Total
	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M	US$M
Gross Revenue	0	0	0	0	0	657	891	981	910	986	5,331	11,469	31,835	53,061
Realisation Costs	0	0	0	0	0	63	86	95	88	95	516	1,109	3,079	5,132
Net Sales Revenue	**0**	**0**	**0**	**0**	**0**	**593**	**805**	**886**	**822**	**891**	**4,816**	**10,360**	**28,756**	**47,929**
Mining	0	0	0	0	0	99	132	147	158	158	884	2,343	6,556	10,476
Processing	0	0	0	0	0	46	52	52	52	52	259	517	2,096	3,125
Smelting	0	0	0	0	0	32	37	39	37	39	204	427	1,389	2,203
Tailings	0	0	0	0	0	1	1	1	1	1	7	13	56	83
Other	0	0	0	0	0	37	37	34	30	27	111	220	1,120	1,616
Total Site Operating Costs	**0**	**0**	**0**	**0**	**0**	**216**	**258**	**274**	**278**	**277**	**1,464**	**3,520**	**11,216**	**17,503**
Operating Surplus/(Deficit)	**0**	**0**	**0**	**0**	**0**	**378**	**547**	**613**	**544**	**614**	**3,352**	**6,840**	**17,540**	**30,427**
Indirect Costs (incl. Depreciation)	0	0	0	0	0	972	149	156	125	113	602	523	2,066	4,706
Net Profit Before Income Tax	**0**	**0**	**0**	**0**	**0**	**(594)**	**398**	**457**	**419**	**501**	**2,750**	**6,317**	**15,474**	**25,721**
Income Tax	0	0	0	0	0	0	0	80	126	150	825	1,895	4,714	7,790
Net Profit After Income Tax	**0**	**0**	**0**	**0**	**0**	**(594)**	**398**	**376**	**293**	**351**	**1,925**	**4,422**	**10,760**	**17,931**
Capital Expenditure	(20)	(30)	(496)	(645)	(702)	(156)	(62)	(99)	(35)	(19)	(270)	(274)	(1,741)	(4,549)
Depreciation & Working Capital	(2)	(1)	(37)	(12)	(43)	912	118	134	131	93	519	480	1,965	4,257
Net Cash Flow After Tax	**(21)**	**(31)**	**(533)**	**(657)**	**(745)**	**162**	**455**	**411**	**389**	**425**	**2,174**	**4,628**	**10,984**	**17,640**

23.0 ADJACENT PROPERTIES

There are no adjacent properties relevant to this Report.

24.0 OTHER RELEVANT DATA AND INFORMATION

Ivanplats has commissioned a number of firms to complete studies in support of an update to the 2012 PEA on the Project as indicated below

- AMC: Overall report coordination, financial modelling, open pit mining, owners costs and general and administration
- AMEC: Geology and resource
- Stantec: Underground mining
- SRK: Geotechnical for underground mining and declines
- Golder Associates: Environmental, social and health impact assessment (ESHIA), tailings dams, hydrology, geohydrology, geotechnical for plant and tailings dam.
- Hatch: Principal engineering consultants, process plant, Supporting infrastructure
- Stucky Limited: Power supply to the mine

No provisional results are available on the update as at the Report effective date. A number of throughput scenarios are under consideration. Potential mining rates could range from 5 Mt/a to as much as 20 Mt/a through the operation of multiple mining areas and a series of production expansions to maximize extraction.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 Mineral Resource Estimate Update

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 (2010).

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-mechanized operation. In addition, the amount of contact dilution related to local undulations in the SMZ has yet to be determined. Ivanplats plans to mitigate these risks with further in-fill drilling.
- Assumptions used to generate the data for consideration of reasonable prospects of economic extraction. Mining recovery could be lower and dilution increased where the dip locally increases on the flanks of the domes. An exploration decline is being considered 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. Location of an exploration decline(s) and the extent of the decline are dependent on further studies.
- Long-term commodity price assumptions.
- Long-term exchange rate assumptions.
- Operating and capital assumptions. Exploitation will require building a greenfields project with attendant infrastructure.
- Metal recovery assumptions. Metallurgical testwork indicates the need for multiple grinding and flotation steps, and variability testwork is yet to be undertaken.
- The fiscal and political regime under which mining operations might occur are uncertain. There is provision within the 2002 Mining Code for the Government to change the 2002 Mining Code and mining rights by decree and a draft 2012 Mining Code has been circulated. There is also a risk that the DRC Government could change the current royalty, duty, and taxation regime.

25.2 2012 PEA

As the 2012 PEA is still considered to be current, the following conclusions are considered to remain reasonable as their underlying assumptions have not changed.

The 2012 PEA examined an initial 5 Mtpa mine production rate without expansions over the mine life. Further study work should be undertaken to optimise the project production rate by considering concentrator and smelter capacities that are matched to the mine production and the available power supply.

The key mine production criterion used for production scheduling in the 2012 PEA was the concentrator feed rate and this resulted in variable concentrate production. Further work should be carried out using a concentrate production rate suited to the smelter capacity. This will allow for a more efficient use of the smelter capital.

Given the significant Mineral Resource tonnage estimated and its large lateral extent, potential mining rates could range from 5 Mtpa to 20 Mtpa, through operating in multiple mining areas and a series of production expansions to maximise extraction of the Mineral Resource.

The 2012 PEA includes capital for an upgrade of the power generating capacity in the region. The power consumption assumed for the 5 Mtpa option in this report does not require the full generating capacity that would result from the capital expenditure. Preliminary work indicates that a concentrator capacity of 7.5 Mtpa may allow a more efficient use of the assumed capital and it is recommended that further study be undertaken to examine this rate as the revised base case capacity.

Selection of the initial plant feed rate to suit the available power capacity may maximise the financial returns from the project.

It is recommended that a further round of study include trade-off analysis of capacity and capital to identify additional optimisations for the Project's production rate.

Ivanplats has undertaken some initial power studies that have identified the availability of initial power capacity. Kamoa contains a large mineral resource and production rate studies have been recommended. Ivanplats will continue to study the power capacity and generation capabilities in the region and use these results for defining production capacity. After developing an understanding of the power options in the region Ivanplats may wish to consider participating in other power projects to secure power for later expansions. Ivanplats have already started in implementing this strategy after having signed an MOU with the Ministry of Energy for the identification of potential sites for a greenfield power plant and have conducted a due diligence operation on available sites to identify a suitable site for this purpose.

Matching of the mining, concentrate production and infrastructure capabilities will require consideration of the ramp up rates and full production rates. For example, relatively small amounts of open pit production are required for the plant feed to allow an earlier start than would be possible from an underground only operation. The work on the open pit mineral resource should be continued as well as options for the declines and underground production rates. The geotechnical mine design criteria for underground mine design, which impacts resource recovery, dilution and mining costs, are mainly based on widely spaced drill holes and for more detailed study this will need to be confirmed by data to be obtained from the underground exposure. Therefore, the mine designs will need to be revised according to the geotechnical recommendations.

26.0 RECOMMENDATIONS

As an update to the 2012 PEA is underway, AMEC has restricted recommendations to a work program consisting of one phase of drilling. The recommended drilling has been broken down by localities within the deposit, and totals 85,231 m, including allocations for exploration, infill, metallurgical, geotechnical, and condemnation purposes. The program is estimated at US$46.5 M.

The work program has been designed to progress knowledge across the Project. AMEC recommends prioritizing the program to benefit development of a mining strategy for the first 10-–15 years of production. This should draw on the September 2012 PEA and development strategies as they emerge from the updated PEA that is in progress.

26.1 Proposed Work Program

Ivanplats is currently undertaking a drilling program that encompasses metallurgical, geotechnical, civil geotechnical and hydrogeological holes. Additional planned holes will be a combination of exploration, Mineral Resource expansion and Mineral Resource delineation to target potential upgrades in Mineral Resource confidence categories and zones of additional mineralized material. Additional engineering drill holes, including metallurgical drilling will be completed. Sterilisation drilling using the AMBL-owned landcruiser-mounted diamond drill rig is planned. Drill holes that will target sources of aggregate for construction purposes are also planned. These will also be drilled using the AMBL-owned core rig.

26.1.1 Kansoko Sud and Makalu

Support for confidence category upgrades will be provided by additional drilling along the edge of the proposed Kansoko Centrale mining block areas and additional infill drilling over the Kansoko Sud and Makalu areas. In-fill drilling in Kansoko Sud and Makalu is intended to test for variability. The program is also designed to support initial estimation of Mineral Resources in areas of new drilling and support potential upgrades of resource confidence categories in areas of existing drilling. In addition, 800 m-spaced extension drilling will test the down-dip extents of the known Kansoko mineralization. A “cross” of closely-spaced drill holes is planned on 100 m centers in Kansoko Centrale to test the variability of the hypogene mineralization and determine the drill spacing that will be required to support resource estimation to higher confidence categories in this area of lower variability. In-fill of the proposed first mining phase area to 800 m depth on 200 m centres is also planned for Kansoko Centrale. The potential to expand the known mineralization will be tested by exploration drilling to the west of the Makalu dome. The drill holes will be sited to test for potential supergene zones and areas of elevated copper grades that may be able to be traced down dip. Drill holes are also planned to target the potential for mineralization down-dip along the Kansoko trend.

26.1.2 Kakula and Kakula NE

Drilling is planned over the Kakula and Kakula NE dome areas to evaluate previously identified soil geochemistry targets which had identified highly-anomalous copper values. The soil anomalies were followed up by four core holes in 2011, which intersected a thick, low-grade mineralized zone. A wide-spaced grid is planned to cover the area and test further for potential targets of Kamoa type mineralization. One hole will be drilled to test the Kamoa Dome stratigraphy as a deep condemnation hole.

26.1.3 Kamoa Ouest and Kamoa Sud

Infill and step-out drilling is also planned on the edges of the current resource model area to support mineral resource estimation and potential conversion of mineralized material to higher-confidence category upgrades. In-fill drilling is also planned in selected areas to potentially support conversion of some or all of the Indicated Resources to Measured Resources. Currently no Measured Mineral Resources have been estimated at Kamoa but during the latter part of 2013, a tightly-spaced grid is planned for the Kamoa Ouest in a potential open-pit area. Additional drilling at Kamoa Sud is planned to cover the area at 200 m spacings.

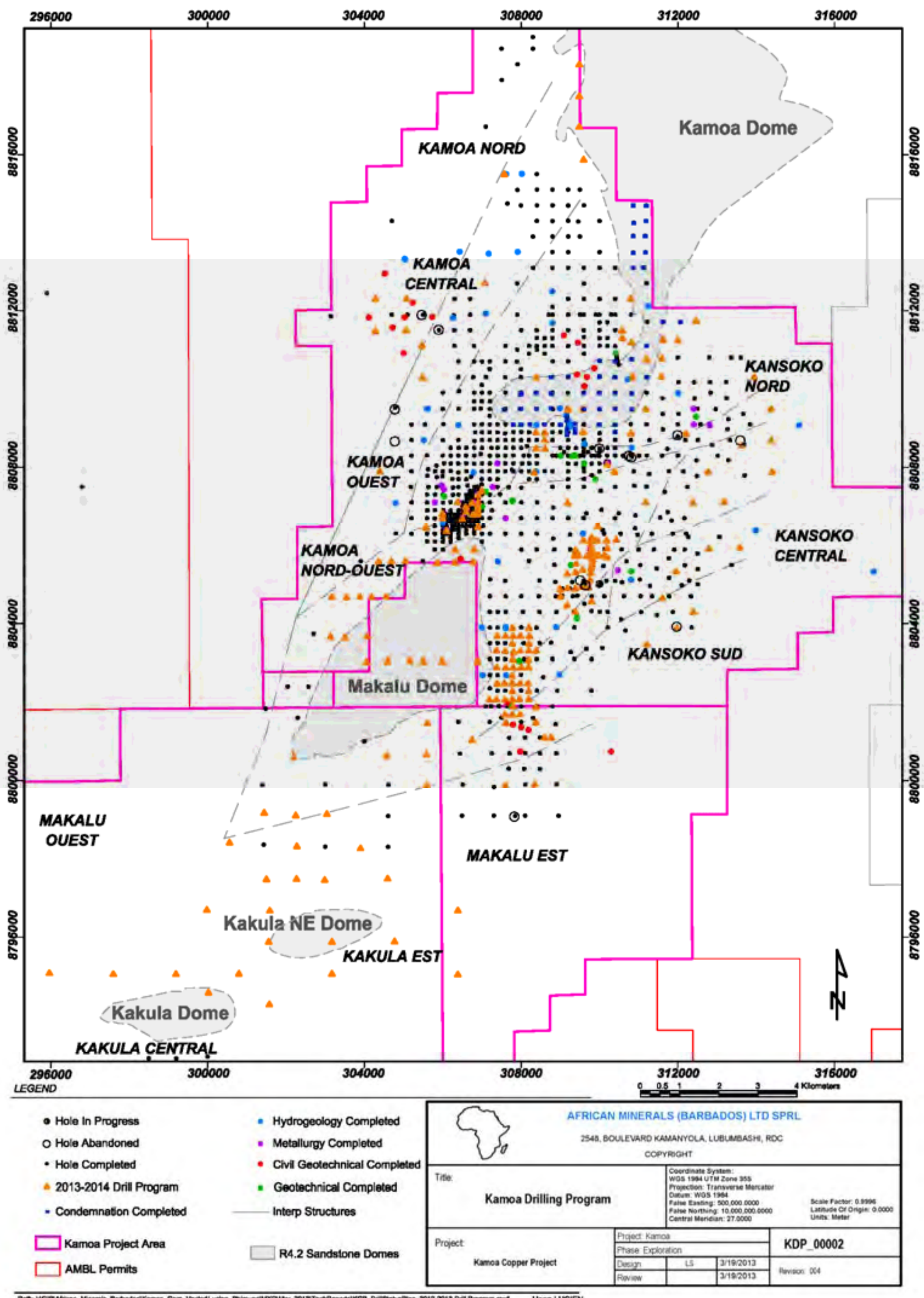
26.2 Program Details

Figure 26-1 is a drill location plan for the Project. On this figure, drill holes completed and in progress as of 11 March 2013 are shown in black; the planned drilling is shown as orange triangles.

Table 26-1 summarizes the committed and planned drilling. Drilling for 2013–2014 is planned to the end of the first quarter of 2014, which coincides with the end of the wet season in the Congo. However, the current plan is for the majority of the drill holes to be completed by the end of 2013 to support more detailed engineering studies. A drilling cost of US$546/m was used for the proposed drilling budget of US$46.5 million.

A number of iterations of Mineral Resource estimate updates would be expected to be undertaken during the drill programs. In the AMEC QPs' opinion, the drilling contemplated is likely, by the end of the first quarter of 2014, to be at a sufficiently close spacing to support completion of detailed engineering studies on the Project.

Figure 26-1: Kamoa Proposed Drill Program



Note: Figure courtesy Ivanplats, modified by AMEC, 2013

Table 26-1: Kamoa Proposed Drill Program

Area	Description	Metres Planned	Method
Various	Metallurgy	3,380	Preliminary variability drilling of 5 holes with 4 wedges per hole.
To be defined	Metallurgy	6,450	Selected variability metallurgical holes on a rough grid within the planned first 5-10 years mining block. Exact locations to be determined.
Kansoko	Metallurgy	5,030	Kansoko close spaced cross of holes to test variability at +/-400m
Kansoko	Infill	14,780	Kansoko infill within planned first 5-10 years mining block at 200m centres.
Makalu Dome	Condemnation	510	Condemnation drilling on Makalu Dome and Kansoko Sud box cut
Various	Geotechnical	5,000	Geotechnical drilling - To be specified by SRK.
mainly Kansoko	Exploration	15,039	Targeting down dip extensions down dip from high grade material
Various	Exploration	5,339	Wide spaced step-out within more exploratory targets on the edges of the current resource
Kakula	Exploration	2,160	Wide spaced exploration drilling at Kakula Exploration with the company's landcruiser rigs for potential supergene open pittable material
Kakula	Exploration	3,180	Wide spaced exploration drilling at Kakula with intension of discovering additional mineralization that may support Mineral Resource estimation
Kansoko Sud	Infill/Metallurgy	14,363	Infill drilling in Kansoko Sud with inclusive metallurgical sampling program
Kakula and Kansoko	Exploration	10,000	Follow up drilling at Kakula and the eastern limits of Kansoko. Hole locations will be determined at a later date.
	TOTALS	***85,231***	

As the planning proceeds, it is recommended that some of the infill drilling planned to support potential conversion of Inferred Mineral Resources to Indicated Mineral Resources be shifted to in-fill drilling planned to support potential conversion of Indicated Mineral Resources to Measured Mineral Resources to support declaration of Proven Mineral Reserves for at least the first five planned years of operation.

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