# INTREPID MINES LIMITED: KITUMBA TECHNICAL REPORT FILED

**Brisbane**, 9 October 2014: Intrepid Mines Limited (ASX: ‘IAU’) (“Intrepid” or the “Company") announces that a report entitled "Kitumba Copper Project Optimised Pre-Feasibility Study NI43-101 Technical Report" dated 25 September 2014, has been filed on SEDAR (www.sedar.com) and on the ASX platform, following the announcement on 28 August 2014, of the proposed merger between Intrepid and Blackthorn Resources Limited (“Blackthorn”). The report was prepared by independent consultants Lycopodium Minerals Pty Limited, in accordance with Canadian National Instrument 43-101 and the JORC Code.

The Kitumba Project (“Kitumba”), situated in west-central Zambia, is Blackthorn’s flagship project. An Optimised Pre-Feasibility Study on Kitumba was completed by Blackthorn in April 2014, and the technical report relates to this study. Should the proposed merger complete as anticipated, Kitumba will be the merged entity’s cornerstone project and management will progress a Definitive Feasibility Study as a priority.

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# INTREPID MINES LIMITED

# KITUMBA COPPER PROJECT OPTIMISED PRE-FEASIBILITY STUDY

## NI 43-101 TECHNICAL REPORT


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# 1.0 SUMMARY

## 1.1 Introduction

Intrepid Mines Limited (Intrepid) has announced a proposed merger with Blackthorn Resources Ltd (Blackthorn). Blackthorns assets include the Kitumba Copper Project (Kitumba) and associated regional exploration assets in Zambia.

The Kitumba Copper Project is part of the larger Mumbwa Project area.

The Mumbwa Project, previously held in joint venture with BHP Billiton (BHPB), is currently owned 100% by Blackthorn through its wholly owned subsidiary Blackthorn Resources (Zambia) Ltd, with BHPB retaining a 2% production royalty following its decision to exit from direct involvement in the Project in 2011.

Intrepid, a Canadian reporting issuer, has requested Lycopodium Minerals Pty Ltd (Lycopodium) to coordinate and prepare an independent technical report on Kitumba, based on the outcomes of the Optimised Pre-feasibility Study (OPFS) completed in May 2014, in accordance with Canadian National Instrument 43-101 requirements and also to report on the associated regional exploration assets.

## 1.2 Property Description and Location

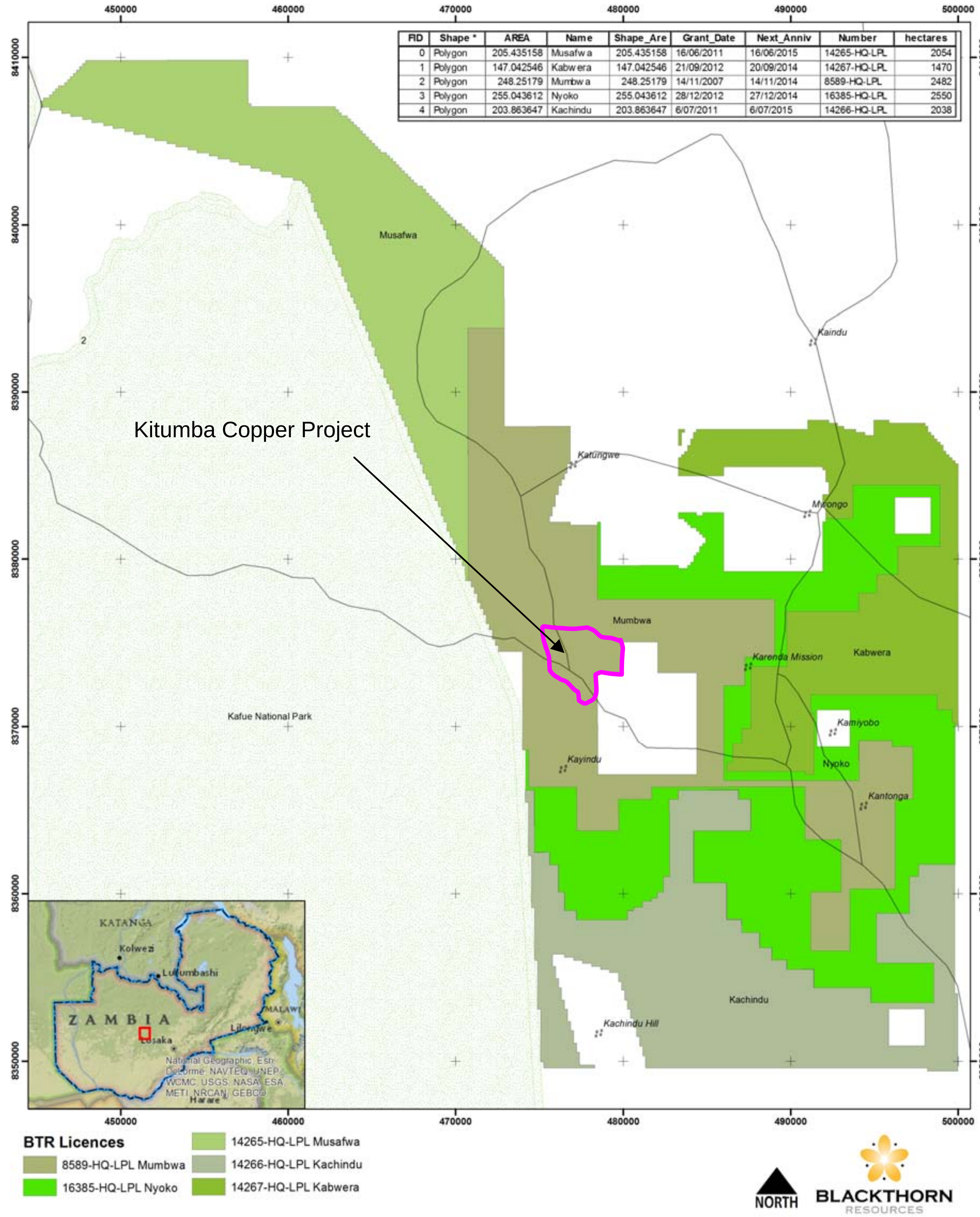
The Kitumba Copper Project is located in west central Zambia, approximately 200 km west of the capital, Lusaka.

Kitumba is currently the main focus of a larger exploration property, the Mumbwa tenement (8589 HQ-LPL), which covers an area of approximately 250 sq km, and is currently being explored for Iron Oxide Copper Gold (IOCG) style mineralisation.

The Mumbwa Project includes five exploration licenses covering approximately 1,059.6 sq km as shown in Figure 1.1.

A preliminary overall layout of the Kitumba site has been included in Appendix 1.

## Figure 1.1 Mumbwa Project Exploration Licenses

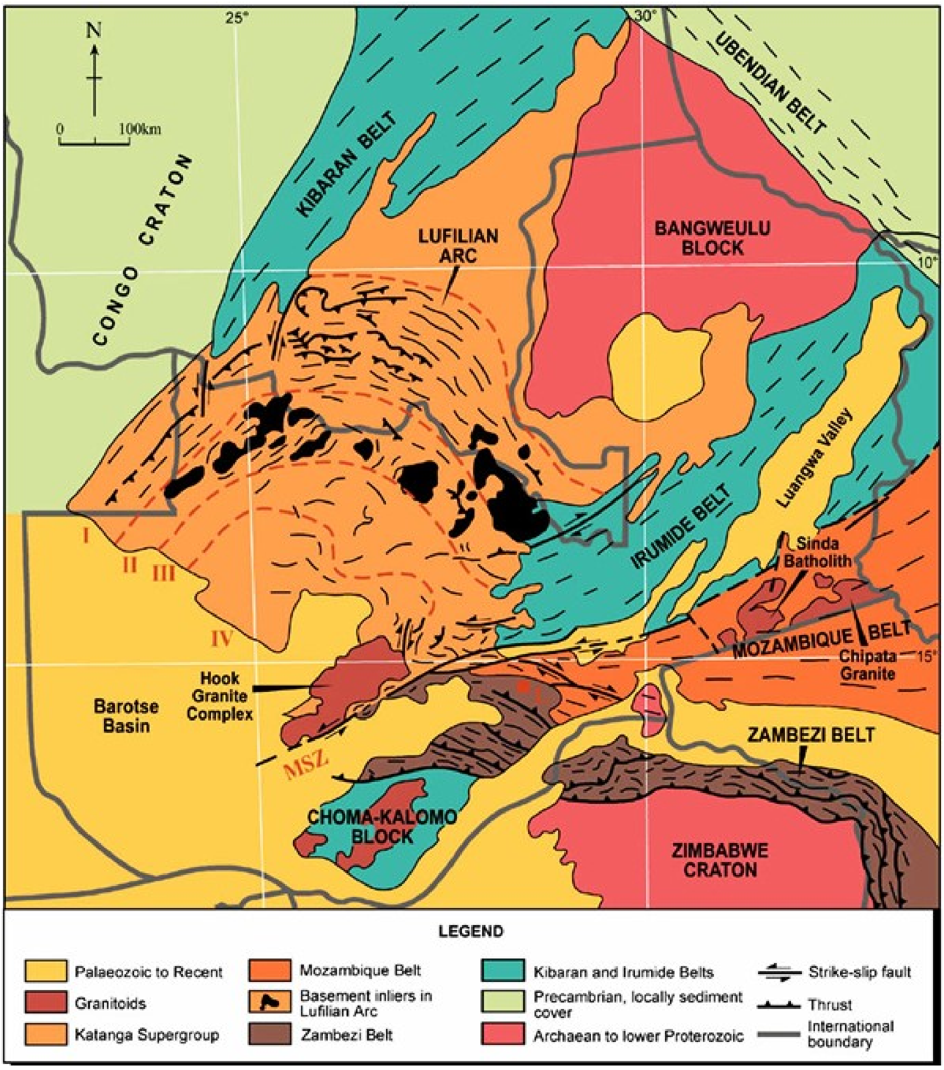


| FID | Shape * | AREA | Name | Shape_Are | Grant_Date | Next_Anniv | Number | hectares |
|---|---|---|---|---|---|---|---|---|
| 0 | Polygon | 205.435158 | Musafwa | 205.435158 | 16/06/2011 | 16/06/2015 | 14265-HQ-LPL | 2054 |
| 1 | Polygon | 147.042546 | Kabwera | 147.042546 | 21/09/2012 | 20/09/2014 | 14267-HQ-LPL | 1470 |
| 2 | Polygon | 248.25179 | Mumbwa | 248.25179 | 14/11/2007 | 14/11/2014 | 8589-HQ-LPL | 2482 |
| 3 | Polygon | 255.043612 | Nyoko | 255.043612 | 28/12/2012 | 27/12/2014 | 16385-HQ-LPL | 2550 |
| 4 | Polygon | 203.863647 | Kachindu | 203.863647 | 6/07/2011 | 6/07/2015 | 14266-HQ-LPL | 2038 |

## 1.3 Geological Setting and Mineralisation

The Mumbwa project area lies within the extreme southern portion of the Neoproterozoic Lufilian Arc. The Lufilian Arc is a large arcuate fold and thrust belt covering north-western Zambia, the southern Democratic Republic of Congo, and eastern Angola. This belt is separated from the Zambezi Belt to the south by the Mwembeshi Shear Zone (MSZ), a prominent crustal-scale eastnortheast trending shear zone extending across Central Zambia. To the northwest, the Lufilian Arc is flanked by the 1.4 - 1.0 Ga Kibaran Belt, and to the southeast by the Mesoproterozic Irumide Belt (Figure 1.2).

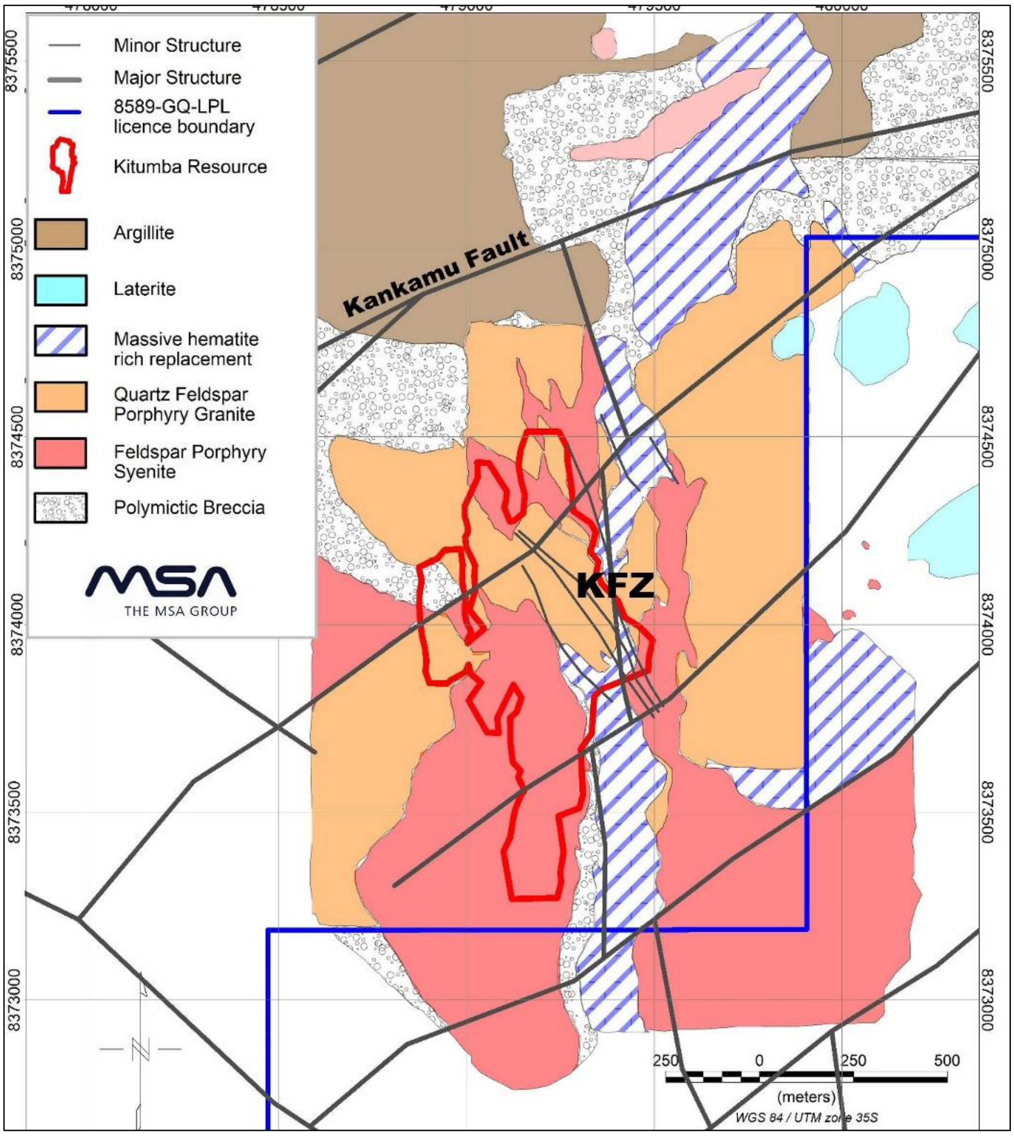
**Figure 1.2 Simplified Geology of Zambia (Modified from Ministry of Mines and Mining Development, 1999)**




The Mumbwa project straddles a regional scale iron-oxide alteration system which is developed along a 26 km long north-northwest to south-southeast trending structural corridor referred to as the Mumbwa Fault Zone (MFZ). Apart from the MFZ, the region is dominated by a northwest and northeast trending conjugate fault set which offsets the MFZ in a number of places.

The Kitumba deposit is considered to be an Iron Oxide Copper Gold (IOCG) type. The Kitumba IOCG deposit is hosted within a hematite-dominated breccia system which is developed along the Kitumba Fault Zone (KFZ) and which outcrops as a prominent north-south trending ridge forming part of the Kitumba Hills. The KFZ is considered to represent a local structure within the broader MFZ.

**Figure 1.3 Geology of the Kitumba Area**




A classic IOCG type alteration system is observed at Kitumba. The system is zoned from north to south, with deeper level magnetite dominated alteration south of Kitumba to the Sugar Loaf deposit (3.8 km southwest of Kitumba) and higher level hematite dominated alteration over Kitumba itself. These correspond to magnetic highs and lows, respectively.

Copper mineralisation at Kitumba comprises a simple hypogene sulphide assemblage that is extensively overprinted by a complex largely redistributed supergene oxide assemblage. Kitumba represents a deeply weathered IOCG system with weathering and oxidation extending up to 500 m depth. Deep weathering is particularly pronounced in the vicinity of the KFZ and zones of high fracture intensity, where leaching of sulphides has generated acid, resulting in a porous and vuggy core, whilst iron is largely preserved. Oxidation of sulphides is noted at significant depths in some holes.

Hypogene mineralisation at Kitumba mainly occurs as disseminated pyrite and subordinate chalcopyrite, associated with stockwork veining and brecciation. Semi-massive concentrations of pyrite ± chalcopyrite are observed in places and are mostly concentrated beneath the richest supergene mineralisation adjacent to the KFZ. Primary sulphides are largely preserved within iron carbonate altered zones, sometimes relatively shallow, where carbonate has acted as a buffer preserving hypogene assemblages.

A significant proportion of the mineralisation occurs in the form of secondary copper minerals within the supergene zone. Supergene enriched material is concentrated from 200 m below surface where it occurs as chalcocite +/- malachite, pseudomalachite, cuprite, native copper, and copper wad. The distribution of secondary copper minerals is related to remobilisation of copper; with secondary copper minerals commonly occurring along fractures and as linings in cavities.

## 1.4 Deposit Types

The Mumbwa district represents an IOCG province that is related to the Hook Granitoid Suite and related rocks that intrude a thick sequence of metasediments along major structures. Numerous copper-gold and iron oxide occurrences are known in the district and are noted on the Geological and Mineral Occurrence Map of Zambia. The Kitumba deposit and its geological setting display similar characteristics to other IOCG deposits.

## 1.5 Exploration

The Kitumba deposit is part of a much larger IOCG system which is potentially mineralized elsewhere along its 26 km north-south strike length. Evidence for this is observed at Kakozhi, 5 km to the northwest of Kitumba. In addition, significant exploration potential exists for the discovery of further IOCG-type mineralisation elsewhere along the continuation of the Mumbwa Fault Zone, particularly at the intersections of northeast trending faults, where soil geochemical anomalies have been defined in previous work.

Work undertaken by Blackthorn within the Mumbwa tenement area includes:

- Remodelling and re-interpretation of the FALCON® airborne gravity gradiometry (AGG), gravity equivalent and magnetic data covering the Kitumba, Mutoya and Mushingashi areas, and generation of new targets.
- Mapping and soil geochemical sampling at Kakozhi.
- An Orion 3D DC-IP-MT survey in the broader Kitumba area.

- A IP-Resistivity survey in the broader Kakozhi area.
- Regional soil geochemical surveys covering previously untested areas elsewhere in the tenement package.
- Diamond drilling.

## 1.6 Drilling

A total of seven phases of drilling have been carried out within the Mumbwa tenement area, with discovery of the Kitumba deposit during the first phase. Subsequent drilling involved mostly resource definition and step-out drilling at Kitumba to determine the limits of the mineralised system. Initial infill drilling was conducted on a 200 m by 200 m grid with subsequent infill down to a current drill spacing of 30 m over the high grade core of the deposit.

Drill testing of satellite targets was mainly based on modelling and interpretation of gravity, magnetic, chargeability and resistivity data. A phase of drilling was also dedicated to testing of a 20 km long linear gravity anomaly to the north of Kitumba.

The two phases of diamond drilling at Kakozki tested integrated 3D geophysical and soil geochemical targets. The results of the drilling at Kakozhi confirm the presence of a mineralised system, with low grade thick intersections associated with the upper oxidised and ferruginised parts of the holes. These may represent zones of leaching and/or reconcentration of copper mineralisation. Potential exists to discover Kitumba-style supergene-enriched mineralisation and hypogene sulphide mineralisation at Kakozhi.

## 1.7 Sample Preparation, Analyses and Security

All core handling, logging, sampling, assaying and QAQC was carried out according to standard operating procedures documented for the project and in accordance with JORC guidelines. An unbroken chain of custody was observed through all phases of the exploration campaign.

Various sample preparation and accredited laboratories have been used over the consecutive phases of work on the Mumbwa project. Samples were analysed by four-acid digest and ICP finish for multi-elements, fire assay and AAS finish for gold, and sulphuric acid leach and AAS finish for determination of acid soluble copper. Ore grade analyses were carried out as routine, once the high grade core of the deposit had been defined.

Best practice QAQC measures were observed including the routine insertion of duplicates, certified blanks and certified reference materials to monitor assay quality. External QC samples in this form comprise approximately 15% of the assay database. A selected 5% of samples were routinely submitted to a second laboratory for check assay. Ongoing QC monitoring and failure criteria were set up and observed, with QC failures being re-assayed together with several samples on either side.

The QP considers the sample preparation, security and analytical procedures employed to be appropriate and adequate for an exploration program of this nature. No aspect of the sample preparation or analysis was conducted by an employee, officer, director or associate of Blackthorn. Sufficient reference materials were used to control analytical processes, appropriate analytical procedures were used that take rock matrices into account and provide acceptable levels of precision, and sufficient checking work was carried out to demonstrate that the data are unbiased and acceptable for use in geological modelling and Mineral Resource estimation.

## 1.8 Data Verification

A number of data verification and QAQC procedures have been applied during drilling, sampling, assay and data management through the various phases of exploration, in order to ensure the veracity of the data. The MSA Group and the QP, Michael Robertson, have been involved in the Mumbwa project since 2006 (pre-discovery), and apart from Phases 3B and 4 managed by BHP Billiton, have been responsible for turnkey exploration management of the project, which has included:

- Development of project-specific standard operating procedures (SOPs) covering all aspects of the exploration work programme and data management.
- Validation of all data on import into the MSA managed central project database.
- QAQC of assay results.
- Verification of mineralised intersections and Competent Person sign-off of News Releases under the JORC Codes (2004 and 2012 Editions) on behalf of Blackthorn Resources Limited.
- Validation of all input data into the various Mineral Resource estimates undertaken by MSA.
- Numerous site visits by the QP between 2006 and 2014.
- An extensive core re-logging programme following Phase 6, in order to ensure standardisation in the database used in the December 2013 Mineral Resource estimate.

Based on the data verification procedures applied over the history of the project, it is the QP's opinion that the quality of the drilling, sampling and assay data are considered to meet or exceed the standards required by JORC and NI 43-101 and that the data are suitable for use in Mineral Resource estimation.

## 1.9 Minerals Processing and Metallurgical Testing

Testwork undertaken for the September 2013 PFS focussed on production of a saleable flotation concentrate and recovery of acid soluble copper from the flotation tails with an atmospheric tank leach. Copper recoveries were however, lower than desired and operating costs were high due to the requirement to truck large quantities of sulphuric acid from the Copperbelt smelters.

It was recognised that the copper mineralogy of the Kitumba deposit is complex and that a different approach to that used in the PFS was needed to maximise copper recovery and to off-set the importation of sulphuric acid, a major contributing factor to the operating costs.

As a consequence, an alternative flowsheet was proposed and an attendant testwork programme developed to validate the revised flowsheet proposal. The testwork programme evaluated sulphide rougher flotation followed by pressure oxidation (POX) leaching (autoclaving) of the sulphide concentrate and hot acid ferric leaching of the rougher tailings. Copper recovery from the combined POX and acid ferric leach slurries will be by solid-liquid separation followed by solvent extraction and electrowinning (SX/EW).

The testwork comprised two major programmes of work to demonstrate the viability of the proposed commercial flowsheet, while a third programme to investigate the amenability of the Kitumba ore to heap leaching was also completed. The first was undertaken on a sample designated Composite 1, representing predominantly supergene material from the early years of the life of mine. This was followed by a second batch testwork programme using a sample designated Composite 2 from drill core deemed more representative of Years 5 onwards and thus likely to contain a significantly lower proportion of secondary copper minerals and a commensurately higher proportion of primary sulphide material. The third programme was performed on Composite 3, blended from selected drill core deemed representative of the initial years of the life of mine and thus predominantly Supergene ore.

The testwork programmes provided adequate information for the preparation of the OPFS process design criteria and mass balance, and to support flowsheet development, preliminary equipment sizing, and estimation of life of mine copper recoveries for the purposes of the study.

## 1.10 Mineral Resource Estimate

The MSA Group (MSA) has completed a Mineral Resource Estimate for the Kitumba deposit.

The effective date of the Mineral Resource is 05 December 2013. It was originally reported under the guidelines of the 2012 edition of the JORC Code and represents an update to the April 2013 estimate completed by MSA, made possible by the Phase 7 infill drilling conducted during 2013. The April 2013 estimate was reported in accordance with the guidelines of the 2004 edition of the JORC Code.

The Mineral Resource was estimated by Ordinary Kriging of total copper, acid soluble copper, cobalt, gold, silver, uranium, iron, manganese, sulphur and density into a three dimensional block model. The estimate was guided by mineralisation domains (leached/low Cu, Supergene and Hypogene) as well as a low grade (0.3%) and a high grade (4.0%) total copper grade shell.

The Mineral Resource estimate has been completed by Mr. J.C. Witley (BSc Hons) who is a geologist with 26 years' experience in base and precious metals exploration and mining as well as Mineral Resource evaluation and reporting. Mr Witley is a Principal Resource Consultant for The

MSA Group (an independent consulting company), is a member in good standing with the South African Council for Natural Scientific Professions (SACNASP) and is a Member of the Geological Society of South Africa (GSSA). Mr. Witley has the appropriate relevant qualifications and experience to be considered a 'Qualified Person' for the style and type of mineralisation and activity being undertaken as defined in National Instrument 43-101 Standards of Disclosure of Mineral Projects (NI 43-101).

The Mineral Resource was classified into Measured, Indicated or Inferred categories in accordance with the guidelines of the 2012 edition of the JORC Code. It should be noted that there are no material differences between the Mineral Resource categories reported herein whether using those defined by JORC (2012) or the CIM Definition Standards on Mineral Resources and Reserves (CIM Definition Standards) adopted by CIM Council on May 10, 2014. The Mineral Resource, Mineral Reserve, and Mining Study definitions as described in the CIM Definition Standards are incorporated, by reference, into National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101).

A summary of the December 2013 Mineral Resource, at a base case cut-off grade of 1% total Cu, is reported by class and mineralisation domain as shown in Table 1.1. It should be noted that the cut-off grades applied are not the result of detailed economic analysis and therefore Mineral Resources that are not Mineral Reserves have not demonstrated economic viability.

There are no known legal, political, environmental, or other risks that could materially affect the potential development of the Mineral Resources.

**Table 1.1 Kitumba Mineral Resource# Above a Cut-Off Grade of 1.0% Cu, as at 5 December 2013**

| Category | Tonnes (Millions) | Cu % | Acid Soluble Cu % | Co ppm | Au g/t | Ag g/t | U ppm | Density t/m$^3$ |
|---|---|---|---|---|---|---|---|---|
| Supergene Domain | | | | | | | | |
| Measured | 6.1 | 3.44 | 1.66 | 205 | 0.04 | 1.3 | 25 | 2.51 |
| Indicated | 15.2 | 2.07 | 1.00 | 180 | 0.03 | 0.9 | 26 | 2.60 |
| M&I | 21.3 | 2.46 | 1.19 | 187 | 0.03 | 1.0 | 26 | 2.57 |
| Inferred | 0.2 | 1.12 | 0.28 | 124 | 0.16 | 0.4 | 22 | 2.66 |
| Hypogene Domain | | | | | | | | |
| Measured | 4.4 | 2.23 | 0.45 | 247 | 0.04 | 1.0 | 21 | 2.86 |
| Indicated | 9.0 | 1.93 | 0.57 | 210 | 0.03 | 0.9 | 32 | 2.83 |
| M&I | 13.4 | 2.03 | 0.53 | 222 | 0.03 | 0.9 | 28 | 2.84 |
| Inferred | 3.9 | 1.39 | 0.23 | 415 | 0.02 | 0.7 | 31 | 2.81 |
| Combined Domain | | | | | | | | |
| Measured | 10.5 | 2.93 | 1.15 | 223 | 0.04 | 1.2 | 23 | 2.66 |
| Indicated | 24.2 | 2.02 | 0.84 | 191 | 0.03 | 0.9 | 28 | 2.69 |
| M&I | 34.7 | 2.29 | 0.93 | 201 | 0.03 | 1.0 | 27 | 2.67 |
| Inferred | 4.1 | 1.38 | 0.23 | 401 | 0.03 | 0.7 | 31 | 2.80 |

**# All tabulated data have been rounded and therefore minor computational errors may occur.**

In order to illustrate the sensitivity of the Mineral Resource to cut-off grade, the Mineral Resource is presented at a variety of cut-off grades in Table 1.2 for combined Measured and Indicated and Table 1.3 for Inferred Mineral Resources.

**Table 1.2 Kitumba Measured and Indicated Mineral Resource# by Cut-Off Grade, as at 5 December 2013**

| Cut Off Grade (Cu%) | Tonnes (Millions) | Cu % | Acid Soluble Cu % | Co ppm | Au g/t | Ag g/t | U ppm | Density $t/m^3$ |
|---|---|---|---|---|---|---|---|---|
| 0.50 | 81.6 | 1.37 | 0.52 | 170 | 0.04 | 0.9 | 28 | 2.67 |
| 1.00 | 34.7 | 2.29 | 0.93 | 201 | 0.03 | 1.0 | 27 | 2.67 |
| 1.40 | 25.1 | 2.72 | 1.16 | 208 | 0.03 | 1.0 | 27 | 2.65 |

**#All tabulated data have been rounded and therefore minor computational errors may occur.**

**Table 1.3 Kitumba Inferred Mineral Resource# by Cut-Off Grade, as at 5 December 2013**

| Cut Off Grade (Cu%) | Tonnes (Millions) | Cu % | Acid Soluble Cu % | Co ppm | Au g/t | Ag g/t | U ppm | Density $t/m^3$ |
|---|---|---|---|---|---|---|---|---|
| 0.50 | 26.2 | 0.79 | 0.15 | 175 | 0.04 | 0.6 | 26 | 2.71 |
| 1.00 | 4.1 | 1.37 | 0.23 | 400 | 0.03 | 0.7 | 30 | 2.80 |
| 1.40 | 1.4 | 1.85 | 0.28 | 231 | 0.03 | 0.5 | 23 | 3.00 |

**#All tabulated data have been rounded and therefore minor computational errors may occur.**

As the mining project is more likely to be an underground project rather than the open pit option considered in April, the base case cut-off grade was increased from 0.5% total Cu in April 2013 to 1.0% total Cu in December 2013 and the April estimates have been re-stated accordingly for comparison purposes in Table 1.4.

**Table 1.4 Comparison of Kitumba Mineral Resource# Above a Cut-Off Grade of 1.0% Cu 5 December 2013 vs 08 April 2013**

| Category | Tonnes (Millions) | Cu % | Au g/t | Ag g/t | U ppm | Density $t/m^3$ | Tonnes (Millions) | Cu % | Au g/t | Ag g/t | U ppm | Density $t/m^3$ |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | December 2013 | | | | | | April 2013* | | | | | |
| Measured | 10.4 | 2.93 | 0.04 | 1.2 | 23 | 2.65 | - | - | - | - | - | - |
| Indicated | 24.2 | 2.02 | 0.03 | 0.9 | 28 | 2.68 | 29.8 | 2.13 | 0.03 | 1.0 | 27 | 2.67 |
| M&I | 34.7 | 2.29 | 0.03 | 1.0 | 27 | 2.67 | 29.8 | 2.13 | 0.03 | 1.0 | 27 | 2.67 |
| Inferred | 4.1 | 1.37 | 0.03 | 0.7 | 30 | 2.80 | 3.5 | 1.39 | 0.04 | 0.4 | 21 | 2.95 |

**#All tabulated data have been rounded and therefore minor computational errors may occur.**
***(Source, MSA 2013)**

The nature, quality, amount and distribution of the data in the Measured and Indicated areas allows for confident determination of the geological framework and to assume continuity of mineralisation allowing for the application of Modifying Factors within a Technical and Economic Study to at least Pre-feasibility level as defined by CIM (2014).

The Phase 7 drilling was successful in upgrading the confidence of a significant portion of the Mineral Resource from Indicated to Measured. Furthermore, the high grade core of the Kitumba deposit was confirmed by a number of drillholes and confidence in the position of the mineralisation state boundaries was enhanced. It is recommended that selective infill drilling is carried out that will further reduce risk in the Mineral Resource.

## 1.11 Mineral Reserve Estimate

The Project Mineral Reserve estimate, classified and reported in accordance with the Canadian Securities Administrators National Instrument 43-101 (NI 43-101) and the corresponding CIM Definition Standards on Mineral Resources and Mineral Reserves, is listed in Table 1.5.

**Table 1.5 Kitumba Mineral Reserve Estimate**

| Item | Tonnes (Mt) | Grade (% Cu) | Metal (kt Cu) |
|---|---|---|---|
| Proven Mineral Reserve | 11.9 | 2.44 | 291 |
| Probable Mineral Reserve | 19.6 | 1.79 | 350 |
| Total Mineral Reserve | 31.5 | 2.04 | 641 |

Mineral Reserves are defined within an underground mine plan generated considering diluted Measured and Indicated Mineral Resources at a 1% Cu COG.

## 1.12 Mining Method

The mining study considered Measured Mineral Resources, Indicated Minerals Resources and Inferred Mineral Resources.

The mining study considered sub-level caving (SLC), a combination of SLC and sub-level open stoping (SLOS) and block caving mining methods. Previous studies considered open pit mining and SLOS.

SLC was selected as the preferred and most suitable mining method for the project.

The factors influencing the mining method selection were:

- The massive geometry of the deposit.
- Near surface mineralisation has been leached of copper.
- A geotechnical assessment that indicates that caving can be induced in areas overlying the orebody.

- Technology – SLC is a mechanized method that allows a high production rate, and low cost.

- Safety – SLC is a non-entry method, where personnel do not enter stoping excavations.

SLC is a top down mining method allowing early production with relatively little predevelopment. In situ ore is progressively blasted in horizontal slices. Rock from above the production horizon caves into the void created by ore extraction, and ore is diluted by the caved rock. Dilution is managed by draw control. As the process progresses, caving propagates upwards and can create surface subsidence.

Figure 1.4 illustrates an isometric view of mine development.

**Figure 1.4 Isometric View of Mine Development**

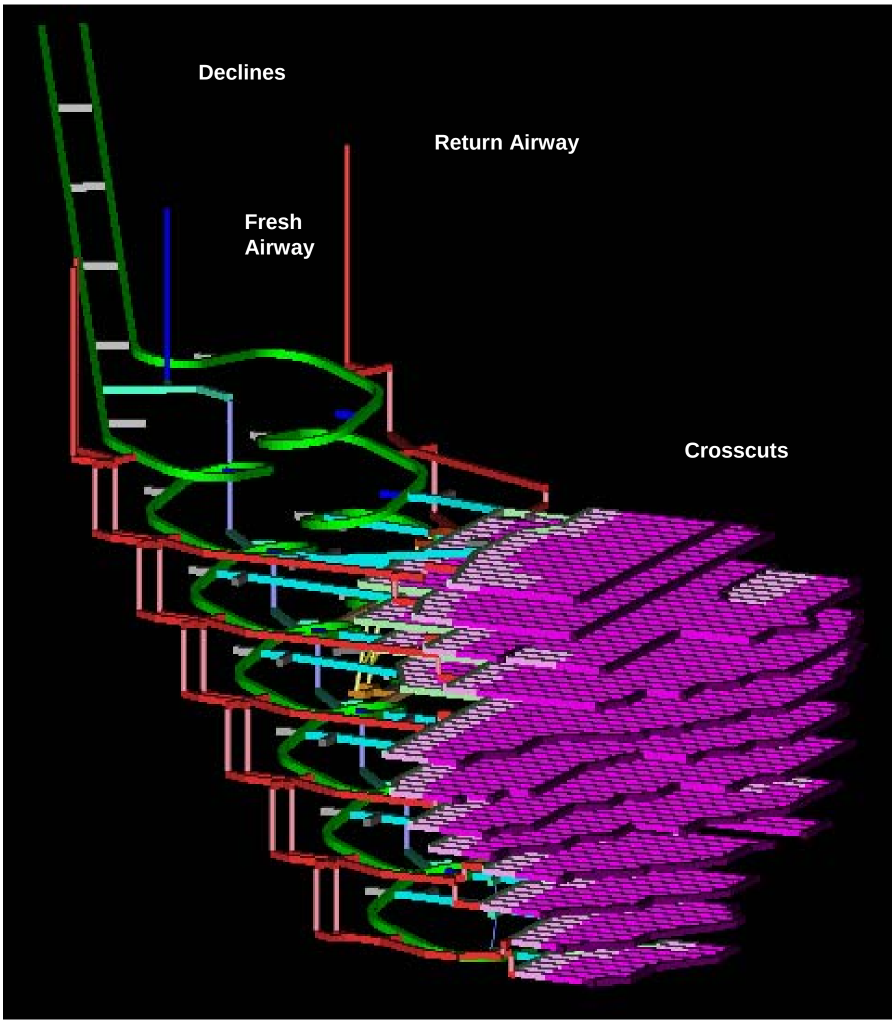
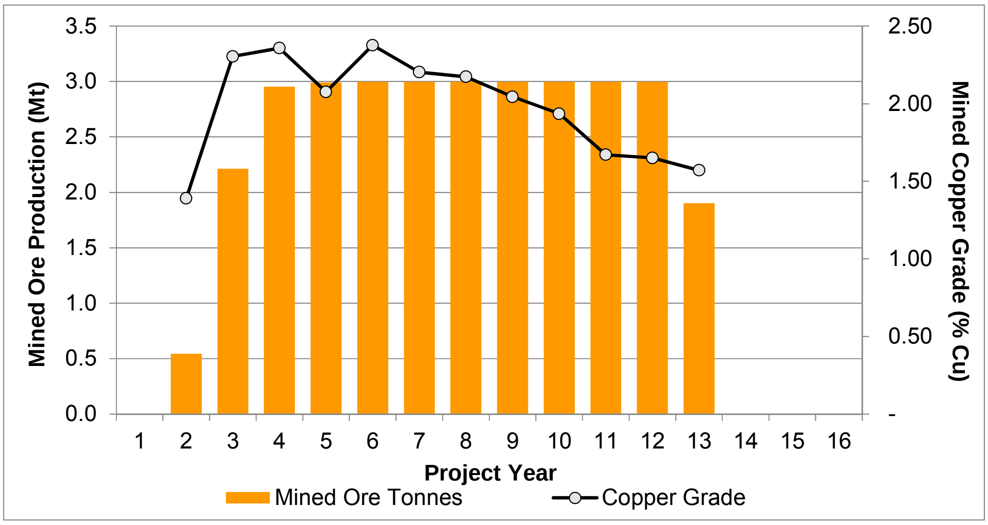


Figure 1.5 shows annual ore production and copper grade.

**Figure 1.5 Annual Ore Production and Copper Grade**




In addition to the mine schedule, AMC prepared a plant feed schedule which was used by Lycopodium to prepare the plant production schedules and predict cash flow.

The plant feed schedule was prepared to:

- Take into account the feed ore throughput ramp up during plant commissioning.
- Blend ore to target a head grade (2.22% Cu) so that the mill is feed 3 Mtpa of ore and 66.7 ktpa of copper metal (60 ktpa recovered copper using a 90% metallurgical recovery).
- Prioritise processing of ore types, based on an AMC applied arbitrary blending classification. The arbitrary blending classifications comprise:
  - Type 1 – Primary sulphide tonnage greater than 1%, copper grade greater than 2.22% copper
  - Type 2 – Primary sulphide tonnage greater than 1%, copper grade less than 2.22% copper
  - Type 3 – Primary sulphide tonnage less than 1%, copper grade greater than 2.22% copper
  - Type 4 – Primary sulphide tonnage less than 1%, copper grade less than 2.22% copper.
- Type 1 and type 2 ore types were prioritised in order to process as much primary sulphide material as possible to assist with acid and heat generation in the processing plant

(shortfall to be made up by importation of concentrates and elemental sulphur), and to reduce residence time in stockpiles (to minimise the effects of oxidation).

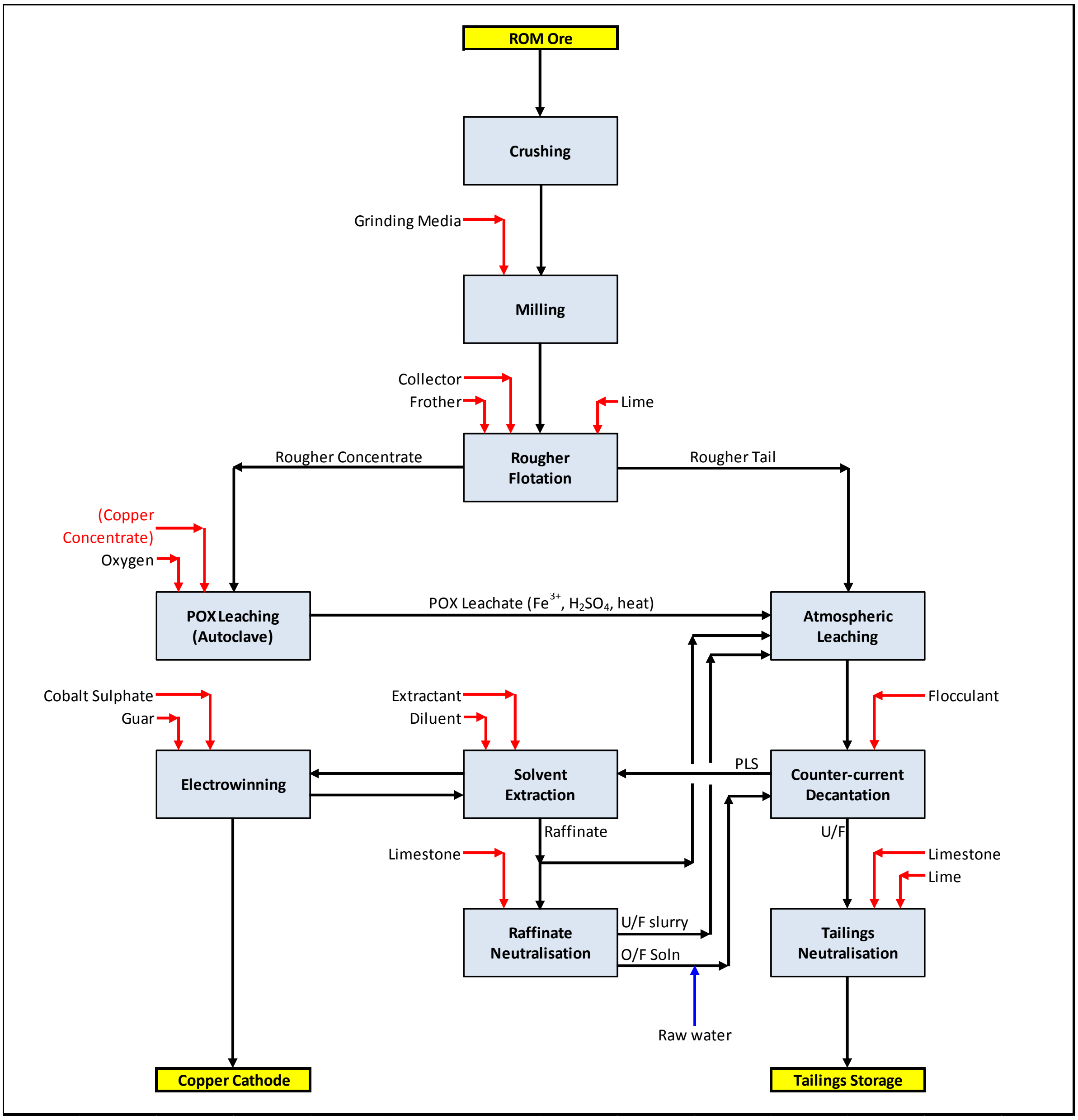
- Minimise stockpile inventories.

## 1.13 Recovery Method

The flowsheet for the Kitumba Copper Project has been designed to process 3 Mtpa of ore to produce nominally 60,000 tpa of copper cathode, although peak copper output of 70,000 tpa may be achieved by increasing the plating current density from a design 310 A/m2 up to 357 A/m2.

The overall block flow diagram for the revised process plant flowsheet is shown in Figure 1.6.

**Figure 1.6 Process Plant – Block Flow Diagram**



The process plant for treatment of the Kitumba ore includes the following unit operations:

- Primary crushing and milling in a SABC circuit to achieve a nominal $P_{80}$ of 150 µm.
- Rougher flotation of the mill product to produce separate concentrate and tails streams.
- Pressure oxidation (POX) leaching of the flotation concentrate in an autoclave to produce sulphuric acid, ferric species, and heat.
- Mixing of the flotation tails with heated raffinate solution and POX leach discharge slurry.
- Leaching of this combined slurry in an atmospheric acid leach circuit to yield a copper rich pregnant leach solution (PLS).
- Separation of the PLS from the barren leach solids in a six stage counter-current decantation (CCD) circuit.
- Extraction of copper from the PLS using solvent extraction.
- Recovery of copper as copper cathode using electrowinning.
- Precipitation and subsequent recycle of copper hydroxide from a bleed stream of solvent extraction raffinate using milled limestone. Recycle of the neutralised raffinate as wash solution in the CCD circuit.
- Neutralisation of the final CCD stage underflow slurry, using limestone and lime, prior to disposal.

A preliminary plant layout drawing has been included in Appendix 2.

The process design criteria are summarised in Table 1.6.

**Table 1.6 Summary Design Criteria**

| Description | Unit | Design Value |
|---|---|---|
| Life of Mine | y | 11 |
| Ore milled, dry | t/y | 3,000,000 |
| Plant Operation | days/y | 365 |
| Plant Operating Hours | h/y | 8,000 |
| Milling Circuit Configuration | - | SABC |
| Milling Circuit Specific Power | kWh/t | 21.4 |
| Milled Product Particle Size, P80 | µm | 150 |
| Flotation Concentrate Mass Pull | % | 7.5 |
| Atmospheric Leach Feed Density | %w/w | 25 |
| Atmospheric Leach Temperature | °C | 60 |
| Atmospheric Leach Residence Time | minutes | 480 |
| Atmospheric Leach Copper Extraction | % | 80 |
| POX Leach Feed S Content | t/y | 37,000 |
| POX Leach Feed Density | %w/w | 20 |
| Oxygen Addition / t S | t/t | 2.5 |
| POX Leach Operating Temperature | °C | 220 |
| POX Leach Oxygen Partial Pressure | kPa | 800 |
| POX Leach Operating Pressure | kPag | 3,110 |
| POX Leach Residence Time | minutes | 60 |
| POX Leach Copper Extraction | % | 98 |
| Copper Production (nominal) | t/y | 60,000 (as cathode) |

A key component for the successful operation of the process flowsheet is the correct regulation of primary sulphide material reporting to the POX leaching circuit, in order to guarantee auto-thermal autoclave operation while generating sufficient reagent (sulphuric acid and ferric ions) and heat for the atmospheric leaching circuit. Consequently, a facility to import an additional source of sulphur (either as pyrite, chalcopyrite, or elemental sulphur) has been included in the design to supplement the POX leach feed if the flotation concentrate sulphur content is low. The Mine Schedule indicates that primary sulphide mineral deficiency will occur mainly within the first four years of production.

Conversely, towards the end of the mine life there will be more than sufficient primary sulphide material to sustain leaching operations. This will result in the need to oxidise a greater mass of sulphur, requiring a larger autoclave and, by association, a larger oxygen plant. The sizing of the autoclave, oxygen plant, and associated POX leaching equipment has taken due cognisance of the variable concentrate feed composition.

## 1.14 Market Studies and Contracts

During the PFS, at a time when the project was forecast to produce a mix of flotation concentrates and cathode copper, Blackthorn commissioned Base Metals Marketing Services Ltd (BMMS) to produce a marketing report to examine the future market for Kitumba's production.

The report was provided in two parts, a Marketing Report for copper concentrates and an Addendum covering the copper cathode market.

With the flowsheet adopted for the OPFS Kitumba becomes a net purchaser of copper concentrates. As Kitumba will be competing with Zambian smelters for concentrates the BMMS report has been used as a basis for determining terms for the purchase of concentrates.

As advised by Blackthorn, a long term copper price of US$3.50/lb has been used for the project Economic Analysis based on information sourced from Wood Mackenzie, Sydney.

## 1.15 Project Infrastructure

Allowance has been made for all infrastructure and facilities necessary to support the on-going mining and processing operation including:

- Water supply from mine dewatering and / or a borefield. A pipeline from the Kafue River has been costed as a 'fall back' position should future hydrogeological studies show it is needed.
- Water storage dam.
- Surface waste rock dump.
- Tailings storage facility.
- Environmental structures for containment of run-off and silt.
- Site access road upgrade.
- A 3 km, 330 kV transmission line from the proposed ZESCO Mumbwa-Kalumbila transmission line with a 2 x 60 MVA transformer step down substation adjacent to the process plant.
- Site wide power distribution.
- Fuel and lubricants storage facility.
- Offices, workshops, warehouses and other buildings including clinic and laboratory.
- Services provided for mine contractors yard and facilities.

- Services for an oxygen plant provided as a 'BOO' facility.
- Potable water supply and sewage treatment.
- 650 person accommodation camp for full operations workforce.
- Temporary accommodation for construction workforce.
- Communications and IT infrastructure.
- Fencing and access control.

## 1.16 Capital and Operating Costs

### 1.16.1 Capital Costs

The overall OPFS capital cost estimate was compiled by Lycopodium. The capital cost estimate reflects the project scope as described in the OPFS report. Mine capital costs were developed by AMC and are included in the estimate table below.

KP provided quantities for a number of items including the tailings storage facility and water storage dam with rates provided by Lycopodium to derive the capital estimate.

All costs are expressed in US dollars ($) unless otherwise stated and based on 1Q2014 pricing. The estimate is deemed to have an accuracy of ±25%.

The various elements of the project estimate have been subject to internal peer review by AMC and Lycopodium and have been reviewed with Blackthorn for scope and accuracy.

The capital estimate is summarised in Table 1.7. The initial project capital cost was estimated at $680.3 million and the maximum cash drawdown to a position where cash flow is positive was estimated to be $696.7 million.

**Table 1.7 Initial Capital Cost Estimate Summary (US$, 1Q2014, ±25%)**

| Main Area | Initial Capital ($M) |
|---|---|
| Mining[1] | 107.8 |
| Construction Indirects | 26.7 |
| Treatment Plant | 266.1 |
| Reagents and Services | 27.3 |
| Infrastructure and Tailings | 67.3 |
| Owners Costs | 42.8 |
| EPCM Costs | 58.8 |
| Contingency | 83.5 |
| **Project Total** | **680.3** |

[1] **Includes AMC capital estimate and accommodation and meals during construction**

The capital cost estimate is subject to the qualifications and exclusions listed in Section 21.1 of the Technical Report.

### 1.16.2 Operating Costs

The overall OPFS operating cost estimate was compiled by Lycopodium. Mine operating costs were developed by AMC and are included in the estimate tables below.

Operating cost estimates are considered to have an accuracy of ±25% and are presented in US dollars ($) based on prices obtained during the first quarter of 2014 (1Q14).

The total operating costs for the Kitumba Copper Project are summarised in Table 1.8.

**Table 1.8 Operating Cost Summary (US$, 1Q14, ±25%)**

| Cost Centre | LOM Cost, $M | $/t Ore | $/t Cu | $/lb Cu |
|---|---|---|---|---|
| Mining | 806.7 | 25.53 | 1,266 | 0.57 |
| Surface and Infrastructure | | | | |
| Operating Consumables | 764.6 | 24.20 | 1,200 | 0.54 |
| Maintenance Materials | 109.7 | 3.47 | 172 | 0.08 |
| Labour | 30.2 | 0.96 | 47 | 0.02 |
| Power | 283.0 | 8.96 | 444 | 0.20 |
| Laboratory | 19.4 | 0.62 | 31 | 0.01 |
| General and Administration | 52.9 | 1.67 | 83 | 0.04 |
| Product Transport | 136.1 | 4.31 | 214 | 0.10 |
| **Total** | **2,202.7** | **69.71** | **3,457** | **1.57** |

The operating cost estimate is subject to the qualifications and exclusions listed in Section 21.2 of the Technical Report.

## 1.17 Economic Analysis

### 1.17.1 Variation from OPFS Evaluation

The mine schedule developed for the OPFS included a small quantity of material (139 kt with 918 t contained copper) which is classified as Inferred Resources. NI 43-101 Rules and Policies (Section 2.3(1)(b)) require that "An issuer must not disclose the results of an economic analysis that includes or is based on inferred mineral resources…" therefore the material falling into the inferred category has been removed from the processing schedule and the Cash Flow Modelling undertaken for the OPFS repeated for the purposes of this Technical Report. The re-classification of this material as waste rather than mill feed does not materially affect the proposed mine plan or mining costs.

### 1.17.2 Cash Flow Model Outcomes

The cash flow model reports:

- All costs in real US Dollars ($) exclusive of escalation or inflation.
- A net present value (NPV) at an 8% discount rate; and
- An internal rate-of-return (IRR) based on after-tax net cash flows.

The outcomes are summarised in Table 1.9.

**Table 1.9 Project Economic Analysis Summary**

| | | |
|---|---|---|
| Revenue from copper (based on $3.50/lb) | 4,909.4 | $M |
| Total cash cost excluding royalties (C1) | 1.57 | $/lb Cu |
| Total cash cost (including royalties) | 1.81 | $/lb Cu |
| All-in cost * | 1.89 | $/lb Cu |
| Capital expenditure (Life-of-Mine) | 796.2 | $M |
| Initial capital investment (excl working capital) | 680.3 | $M |
| Peak funding | 696.8 | $M |
| Deferred and sustaining capital | 115.9 | $M |
| Plant and equipment salvage | 50.0 | $M |
| Closure cost | 33.5 | $M |
| Pre-Tax Economics | | |
| Free cash flow after cost allocation (undiscounted) | 1,628.6 | $M |
| Internal rate of return (IRR) | 25.3 | % |
| Project NPV (discounted at 8.0%) | 694.7 | $M |
| Payback period | 3.4 | Years |
| After-Tax Economics | | |
| Free cash flow after cost allocation (undiscounted) | 1,202.0 | $M |
| Internal rate of return (IRR) | 21.2 | % |
| Project NPV (discounted at 5.0%) | 458.7 | $M |
| Payback period | 3.5 | Years |

*** Total cash cost, including sustaining and deferred capital.**

The basis for the project economic analysis is outlined in Section 23 of the Technical Report along with the qualifications, exclusions and assumptions on which it is based.

## 1.18 Other Relevant Data and Information

### 1.18.1 Project Implementation

The approach to project implementation outlined in Section 24.1 was used as the basis for the preliminary implementation schedule and the build up of the capital cost estimates. The approach to be adopted for project implementation will be developed further during the definitive feasibility study with the preparation of a preliminary Project Execution Plan (PEP).

As the development of the boxcut, portal, decline, production levels and underground infrastructure is on the critical path for achieving earliest possible copper production, the early works will be undertaken by an experienced underground mining contractor under the direction of senior Kitumba mine operations personnel.

Once full project 'go-ahead' is approved, a duration of 25 months was estimated to first ore to mill. Ramp up to 75% production was estimated to take three months with a further six months to reach nameplate capacity.

Several items and activities are potentially on or close to the schedule critical path for design and construction including mine development and supply and installation of long lead mechanical and electrical equipment items for the process plant. Their criticality should be investigated further during the DFS.

The establishment of the power supply from the Zambian national grid while not currently identified as a critical path item has the potential to influence the schedule as it is dependent on activities not completely within the projects control. This has been identified as a potential risk to the schedule. Potential sources of delay include establishment of a suitable easement for the power line, completion of commercial negotiations with ZESCO as well as subsequent technical approvals and agreements regarding responsibility for tendering and managing the supply and installation of the HV power line and switchyards.

### 1.18.2 Operations

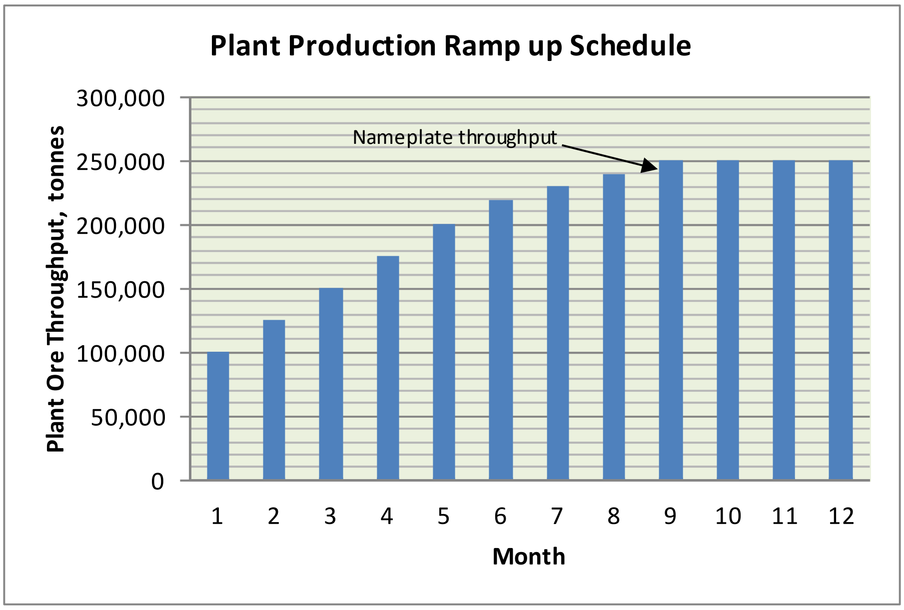
The current operating cost estimate is based on a preliminary organisational structure that is considered to be appropriate for an operation of this scale and type in the region.

Mine development and operations will be undertaken by an experienced mine contracting company under the direction of an owners technical and operations management team assisted, as required, by external consultants and service providers.

Surface operations and administration roles will predominantly be filled by Kitumba staff. Exceptions to this include the operation and maintenance of the cryogenic oxygen plant, which will be supplied and operated under a Build Own Operate contract, and the maintenance and operation of the accommodation camp which will be contracted to a specialist service provider. Consideration will be given to contacting other non-core functions such as site security at an appropriate time.

The plant ramp up schedule has been developed to reflect the complexity of the processing facility, using ramp up data from similar commercial operations as a basis. It is estimated that nameplate plant ore throughput (250,000 t/month) will be achieved nine months after ore is first introduced. The first month will process an estimated 100,000 tonnes of ore, increasing at the rate of 25,000 tonnes per month over the next five months. Month 6 will process 220,000 tonnes, increasing by 10,000 tonnes per month for the next three months, until nameplate ore throughput is achieved at the end of Month 9. The process plant ramp up schedule is illustrated in Figure 1.7.

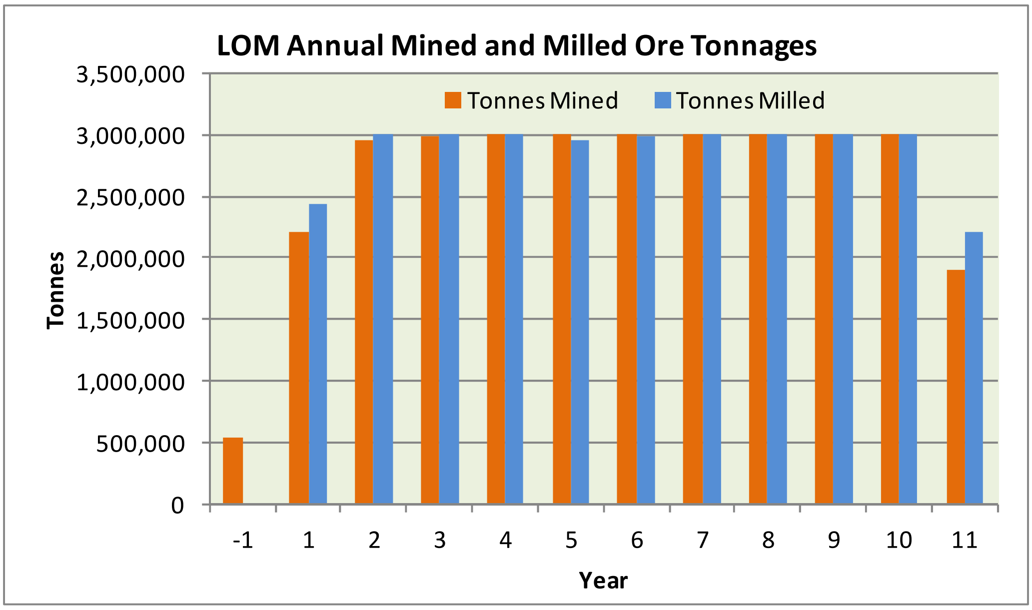
**Figure 1.7 Plant Production Ramp-up Schedule**




The LOM mine production will be 31.6 Mt. Ore mining will commence one year prior to ore being fed to the processing plant, i.e. ore mining will start in Month -11 and plant feed will commence in Month 1.

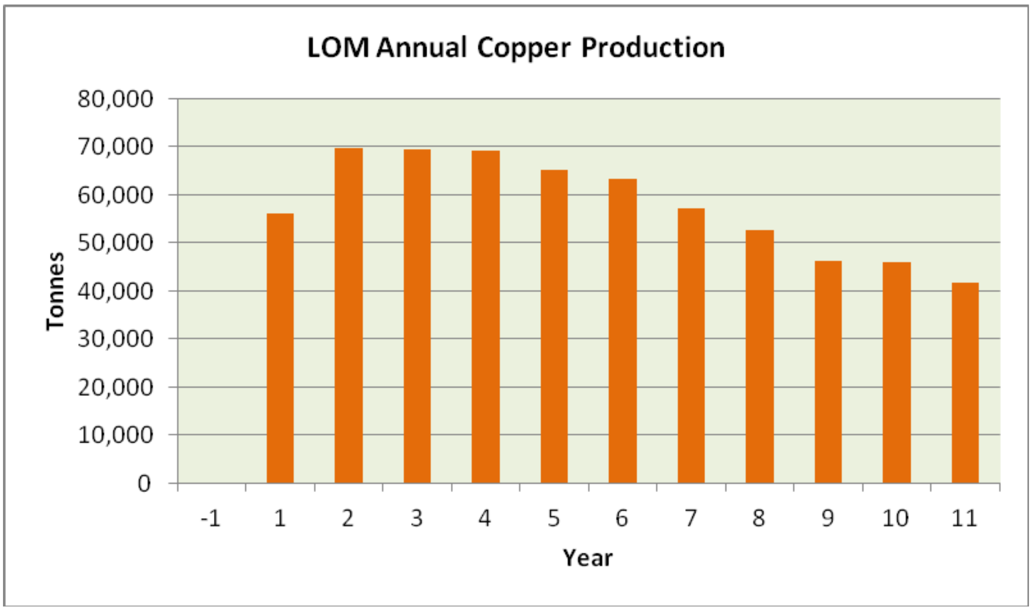
An annual comparison of the LOM tonnage mined and tonnage milled is shown in Figure 1.8.

**Figure 1.8 LOM Annual Ore Production Schedule**




The LOM copper production, including the contained copper recovered from the imported flotation concentrate, is presented on an annual basis in Figure 1.9. The copper production peaks in Year 2 and 3 at circa 70 ktpa, primarily due to the higher copper head grade associated with the supergene material and also partly due to the copper contribution from the imported concentrate.

**Figure 1.9 LOM Copper Production**



## 1.19 Interpretation and Conclusions

Exploration, investigations and studies to date have identified that an opportunity exists for the development of a mine and processing plant at Kitumba, Zambia, to produce nominally 60,000 tpa of copper cathode for an eleven year mine life. Both the proposed mine and processing plant utilise techniques and technologies that are well established in the mining / processing industry.

The proposed operation will be based on exploitation of the Kitumba copper resource with modest quantities of copper concentrates purchased from other operations in Zambia to supplement the process plant feed in certain years.

Zambia is considered to be a mining friendly location with copper mining in particular well established, albeit not in the immediate vicinity of the proposed development.

Sufficient confidence in the outcomes of multi-disciplinary investigations and studies at a pre-feasibility study level of confidence exists to warrant more detailed investigations and the development of a definitive feasibility study to support a final investment decision.

## 1.20 Recommendations

Additional site works, drilling and sampling, laboratory testwork and investigations and collection of data will have to be undertaken to fill gaps in the existing knowledge base.

Included in these activities are the following:

- Diamond drilling to obtain additional metallurgical samples.
- Hydrogeological work for mine design quantification of mine dewatering requirements and identification of supplementary water resources.
- Preliminary geotechnical work on the proposed plant site and sites of major infrastructure items.
- Additional batch metallurgical variability testwork.
- Metallurgical pilot plant investigations.
- Additional mine geotechnical data collection and analysis.

The information generated from these activities will be combined with the output of existing and additional engineering and estimating work to complete a definitive feasibility study. This will complete the definition of project scope, provide ±10-15% capital and operating cost estimates for use as the basis for control budgets, and develop a preliminary execution plan and schedule for the project.

In conjunction with the OPFS consultants, Blackthorn has prepared and costed activities required to complete the definitive feasibility study and take the project through to an investment decision milestone. The activities and budget estimate are shown in Table 1.10.

**Table 1.10 Definitive Feasibility Study Budget Estimate**

| Activity | Total AUD Revised Budget 30 May 2014 |
|---|---|
| Study management inc cost estimate and report writing | 1,860,000 |
| Metallurgical testwork | 820,000 |
| Pilot Plant | 3,700,000 |
| Tailings inc waste rock and geochem | 346,000 |
| Hydrogeology and Geochemical (Exc drilling) | 683,144 |
| Environmental | 620,527 |
| Mine design and scheduling | 365,000 |
| Geotechnical Assessment | 619,000 |
| Drilling Costs (12,500 m total) | 6,500,000 |
| BTR project management & corporate inc risk assessment | 775,000 |
| Peer reviews | 100,000 |
| Project expenses incl 3rd party site visits | 120,000 |
| Marketing | 120,000 |
| Contingency (20%) | 2,815,279 |
| **Total For Final Investment Decision** | **19,443,950** |

Costs shown in Table 1.10 are in Australian Dollars and convert to approximately US$18.1 million at an exchange rate of AUD1.0 = US$0.93.

It is recommended that work continues on activities related to the completion of a definitive study. In particular:

- Mining and geotechnical studies are completed to provide additional confidence in the mine design
- Testwork and pilot programs proceed to demonstrate proof of concept for the plant flowsheet
- Hydrogeology studies are completed to reduce uncertainty regarding water volumes expected to be encountered during mining and to firm up the inputs into the overall site water balance
- Discussions with ZESCO proceed with a view to signing a MoU for power supply for the future project

- Investigations and discussions are held with third parties within Zambia to increase confidence in the availability of key bulk reagents such as limestone and lime and terms for the purchase of copper concentrates to supplement feed to the POX autoclave.

# 2.0 INTRODUCTION

Intrepid Mines Limited (Intrepid) has announced a proposed merger with Blackthorn Resources Ltd (Blackthorn). Blackthorns assets include the Kitumba Copper Project and associated regional exploration assets in Zambia.

Intrepid, a Canadian reporting issuer, has requested Lycopodium Minerals Pty Ltd (Lycopodium) to coordinate and prepare an independent technical report on Kitumba, based on the outcomes of the Optimised Pre-feasibility Study (OPFS) completed in May 2014, in accordance with Canadian National Instrument 43-101 requirements, and also to report on the associated regional exploration assets.

Lycopodium Minerals Pty Ltd is a subsidiary of Lycopodium Limited, an Australian listed public corporation, and has provided engineering and project management services to the international mining industry for over 20 years. The Perth, Western Australia, office of Lycopodium Minerals undertook the preparation of this report.

Sections of this report were co-authored by Qualified Persons from AMC Consultants Pty Ltd (AMC), Perth, The MSA Group (MSA), Johannesburg, and Knight Piésold Consulting (KP), Perth.

AMC is a mining consultancy, providing services exclusively to the minerals sector. Wholly owned by its employees, AMC is headquartered in Melbourne and has offices in Adelaide, Brisbane, Perth, Toronto, Vancouver and Maidenhead in the UK. The Perth office of AMC contributed to the preparation of this report.

MSA is a provider of exploration, geology, mineral resource and reserve estimation, mining and environmental consulting services to the international minerals industry, and has been providing such services since 1983. Based in Johannesburg, South Africa, MSA has had involvement in the Kitumba Copper Project since 2006.

KP is an international firm of consulting engineers and scientists with over 90 years of experience in the mining and power sectors. The firm has over 25 offices worldwide and specialises in environmental, civil and geotechnical engineering associated with mining projects in a wide range of commodities and geographical locations. The Australian offices of KP conducted studies for tailings management, surface water management and groundwater evaluations for the project.

This report is based on information provided by Blackthorn including documents, data and reports compiled by Blackthorn management and technical staff and previous reports by other independent experts (refer Section 3.0).

Lycopodium has not undertaken a site visit to Kitumba but the country / region is well known and Lycopodium has attended multiple visits to the region in association with other project briefs. Personnel from AMC, MSA, and KP have all undertaken one or more site visits including visits by relevant Qualified Persons as detailed on the Certificates provided.

This report is provided to Intrepid in connection with its proposed merger with Blackthorn and its obligations under NI 43-101 requirements and should not be used or relied upon for any other purpose.

# 3.0 RELIANCE ON OTHER EXPERTS

In preparing the OPFS and this Technical Report the authors have relied on information supplied by the following:

- Blackthorn Resources, Level 5, Suite 502, 80 William Street, Sydney, NSW, Australia for information regarding the ownership and legal standing of the project.
- Pells Sullivan Meynink, 35 Jeays Street, Bowen Hills, QLD, Australia: report Kitumba Copper Project Pre-feasibility Study Phase 7 Geotechnical Assessment included in Appendix 4.1 of the Optimised Pre-feasibility Study.
- AGES Gauteng, 309 Glenwood Road, Pretoria, RSA: Kitumba Copper Gold Project: Hydrogeological specialist investigation and water management plan: Report number AS-R-2013-09-10 included in Appendix 8.1 of the Optimised Pre-feasibility Study.
- RPS, 38 Station Street, Subiaco, WA, Australia: report Kitumba Groundwater and Surface Water Assessment – Optimised PFS Study included in Appendix 8.2 of the Optimised Pre-feasibility Study.
- REDE Engineering and Management Solutions (Pty) Ltd, Quintin Brand Street, Pretoria, RSA: report BTR Kitumba Project: Bulk Water Value Engineering included in Appendix 8.3 of the Optimised Pre-feasibility Study.
- ZESCO (Zambia Electricity Supply Corporation): report Blackthorn Resources Kitumba Mine Power Supply Project included in Appendix 10.2 of the Optimised Pre-feasibility Study.
- AGES Gauteng, 309 Glenwood Road, Pretoria, RSA: Environmental Pre-feasibility Study Report for the Proposed Kitumba Copper Project, Mumbwa District, Central Province, Zambia, Report number AS-R-2013-06-20 included in Appendix 12.1 of the Optimised Pre-feasibility Study.
- Base Metals Marketing Services Ltd, Marketing Report Mumbwa Copper Concentrates and Addendum 1 (June 2013) included in Appendix 14.1 of the Optimised Pre-feasibility Study.

In all cases the content of the contributory information and reports have been reviewed and are believed to be appropriate and factually correct for the purposes of the OPFS and this Technical Report.

# 4.0 PROPERTY DESCRIPTION AND LOCATION

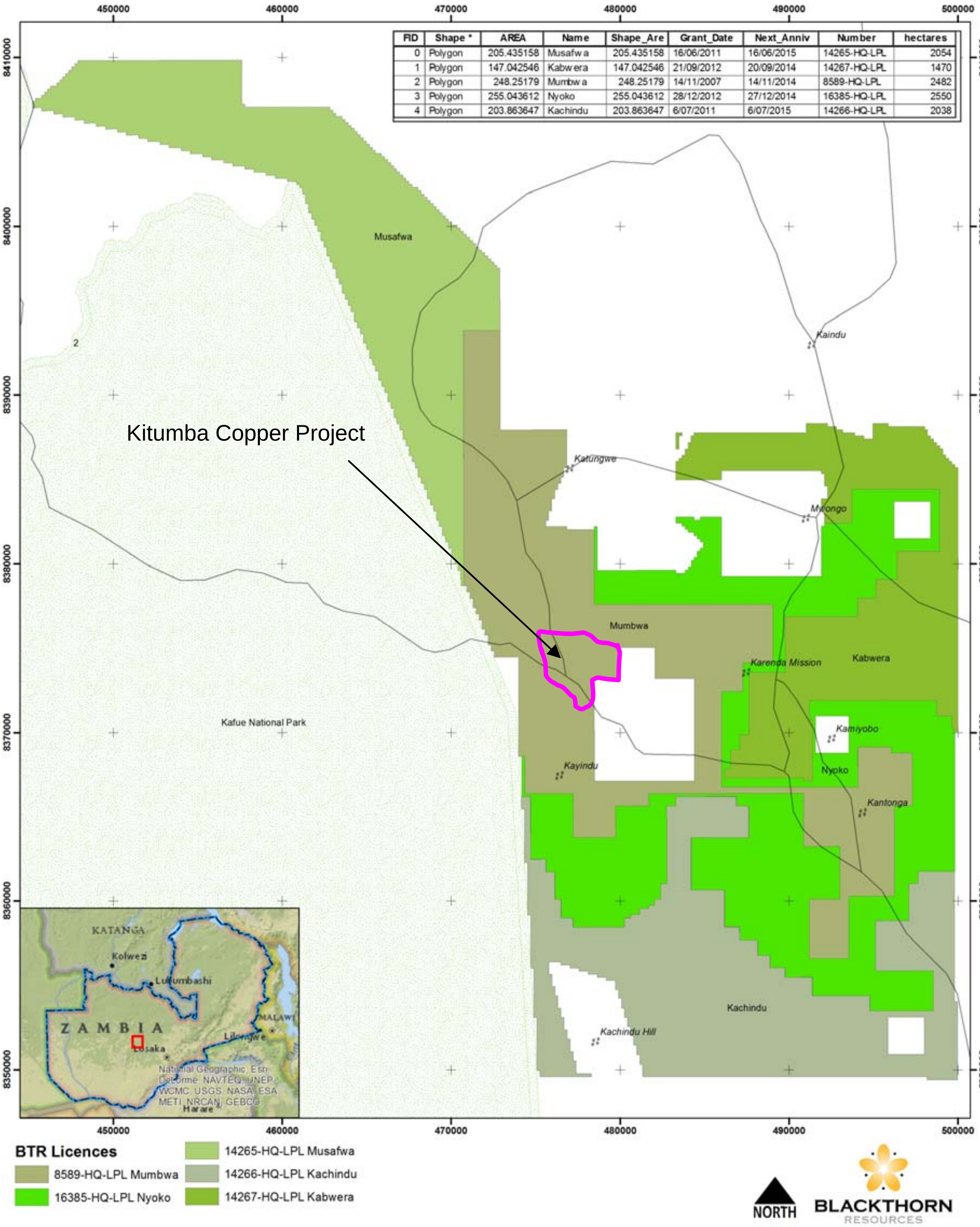
The Kitumba Copper Project is located in west central Zambia, approximately 200 km west of the capital, Lusaka.

Kitumba is currently the main focus of a larger exploration property, the Mumbwa tenement (8589 HQ-LPL), which covers an area of approximately 250 sq km, and is currently being explored for Iron Oxide Copper Gold (IOCG) style mineralisation.

The Mumbwa Project includes five exploration licenses covering approximately 1,059.6 sq km as shown in Figure 4.1.

A preliminary overall layout of the Kitumba site has been included in Appendix 1.

**Figure 4.1 Mumbwa Project Exploration Licenses**




| FID | Shape * | AREA | Name | Shape_Are | Grant_Date | Next_Anniv | Number | hectares |
|---|---|---|---|---|---|---|---|---|
| 0 | Polygon | 205.435158 | Musafwa | 205.435158 | 16/06/2011 | 16/06/2015 | 14265-HQ-LPL | 2054 |
| 1 | Polygon | 147.042546 | Kabwera | 147.042546 | 21/09/2012 | 20/09/2014 | 14267-HQ-LPL | 1470 |
| 2 | Polygon | 248.25179 | Mumbwa | 248.25179 | 14/11/2007 | 14/11/2014 | 8589-HQ-LPL | 2482 |
| 3 | Polygon | 255.043612 | Nyoko | 255.043612 | 28/12/2012 | 27/12/2014 | 16385-HQ-LPL | 2550 |
| 4 | Polygon | 203.863647 | Kachindu | 203.863647 | 6/07/2011 | 6/07/2015 | 14266-HQ-LPL | 2038 |

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

## 5.1 Transport Links

Zambia has a well established mining industry and transport links from the Copperbelt in the north, and the capital Lusaka, to ports in East Africa and ports and industry in Southern Africa are well established.

The M9 from Mumbwa is the main regional road in the area linking to international routes via Lusaka. The road is tarred and in good condition.

Traffic on the outskirts of and through Lusaka is significantly congested however there are plans by the Roads Development Agency (RDA) to develop a bypass. Traffic through Mumbwa is significantly more prominent than in the surrounding rural areas. Existing vehicle traffic volumes on local roads such as the D181 between Mumbwa and the project site are limited. Thus, in terms of existing transportation infrastructure, challenges presented to the project nearer the mine site relate mainly to adequacy of infrastructure (poor road conditions) while closer to Lusaka, the main concern is related to time lost due to traffic congestion.

The D181 from Mumbwa to the project site is a dirt road in poor condition. In some areas, tall grass grows right up to the edge of the road, hampering visibility. During the rainy season, much of the area is not easily negotiable. Local inhabitants use the roads for access mostly by foot or bicycles.

Unrelated to the development of the Kitumba Project, the Zambian government has announced its intention to upgrade the public portion of this road, and the basis for the study assumes that this will be done at no cost to the project and in time for construction traffic. Interim upgrading of the road for construction traffic may be required if the D181 upgrade is delayed.

Kenneth Kaunda International Airport (formerly Lusaka International Airport) provides scheduled passenger and airfreight links to Europe (KLM, at least until October 2014), South Africa (SAA), Dubai (Emirates) and regional capitals.

## 5.2 Climate

Zambia is situated in a subtropical climate generally described as pleasant tropical, but seldom unpleasantly hot outside of the valleys, and comprises three distinct seasons, namely the cool dry season (April - May to August), the hot dry season (August to November); and the rainy season or warm wet season (November to April).

The project site is located in a medium-rainfall area on what is termed the 'central plateau' with rainfall between 800 to 1,000 mm per annum. Lightning is observed regularly. The altitude ranges from 1,000 m to 1,500 m above sea level and the climate is described as mild with temperatures rarely exceeding 35°C. Average temperatures are moderated by the height of the plateau.

Maximum temperatures in the cool season range from 15°C to 27°C while minimum temperatures vary from 6°C to 10°C. Occasional frost occurs on calm nights in valleys and hollows which are sheltered from the wind. Prevailing winds during this season are dry south-easterlies and cause cloudy to overcast conditions. During the hot season maximum temperatures may range from 27°C to 35°C.

## 5.3 Local Resources

Mining is central to the Zambian economy and has played a key role in the social and economic development of the country. Zambia is predominantly a copper mining country and is the largest copper producer in Africa.

Although the Kitumba site is remote from existing mining operations on the Copperbelt it is anticipated that construction contractors and service industries that have built up around the existing operations will readily support the project during both the construction and operational phases.

The villages of Kaindu, Myombe, Mpundu and Kafucamo are the closest communities to the site. The nearest town of any size to the proposed site is Mumbwa, which is approximately 50 km to the south west.

There is no pool of skilled operating or maintenance personnel close to the project site, although a limited number of local people may be currently working on the Copperbelt and may return to their local villages if the opportunity arises. The project plans to accommodate all operating personnel on site and adopt a rostered 'bus-in, bus-out' system to local and regional centres for Zambian employees.

Expatriate personnel will be recruited regionally (e.g. South Africa) if possible, if not internationally, to fill key roles and provide training and operational support, particularly in the early years.

## 5.4 Site Topography

The regional topography can be described as rolling terrain consisting of hills and valleys. Elevation ranges from approximately 1,100 to 1,400 metres above mean sea level (mamsl). Noticeable topographic high points occur in the area, and the target mining area comprises such a hill (Kitumba Hills). Towards the north of the project area the topography flattens out towards the Kafue River.

The site topography is undulating with four to five hills along the east and north eastern boundary of the site. From the central area the site slopes to the west with a reducing slope flattening out to plains outside of the lease boundary.

The OPFS location for the process plant site is on the central western boundary in close proximity to the decline portal. The TSF is located towards the north western boundary and is positioned to take advantage of the less steep fall so that the embankment heights are minimised.

A preliminary overall layout of the Kitumba site has been included in Appendix 1.

## 5.5 Seismic Hazard Assessment

A preliminary seismic hazard assessment has been carried out for the project, comprising a review of existing project information, historical data from earthquake catalogues and technical publications.

Seismic ground motion parameters have been determined using the results of the probabilistic seismic hazard analyses.

It is recommended that the 1 in 1,000 year earthquake be adopted as the Operating Basis Earthquake (OBE) for the project. For a design operating life of 14 years, the probability of exceedance for the OBE event is 1%. The estimated peak ground acceleration for the 1 in 1,000 year earthquake is 0.13 g.

A design earthquake magnitude of 6.5 is appropriate for the OBE. The TSF and appurtenances are expected to remain functional and any damage from the occurrence of earthquake shaking not exceeding the OBE shall be easily repairable.

It is recommended that the 1 in 10,000 year earthquake be adopted as the Maximum Design Earthquake (MDE) for the TSF, for which the estimated peak ground acceleration is 0.33 g.

A design earthquake magnitude of 7.2 is appropriate for the MDE. Limited deformation of the tailings dam is acceptable under seismic loading from the MDE event, provided that the overall stability and integrity of the facility is maintained and that there is no release of stored tailings or water.

Parameters have also been provided for the seismic design of structures at the project site using the International Building Code (IBC, 2012) and assuming Site Class B (rock) conditions at the site.

Local site foundation conditions will be classified according to IBC requirements to account for potential seismic site responses (amplification of ground motion). A design earthquake magnitude of 6.8 is recommended for geotechnical foundation design of the mine site structures.

# 6.0 HISTORY

Intrepid Mines Limited has announced a planned merger with Blackthorn Resources Ltd. Blackthorns assets include the Kitumba Copper Project tenements in Zambia.

The Kitumba Copper Project is part of the larger Mumbwa Project area (refer Figure 4.1).

The Mumbwa Project, previously held in joint venture with BHPBilliton (BHPB), is now owned 100% by Blackthorn Resources Ltd (Blackthorn) through its wholly owned subsidiary Blackthorn Resources (Zambia) Ltd, with BHPB retaining a 2% production royalty following its decision to exit from direct involvement in the project in 2011.

The Mumbwa area has numerous historic artisanal copper mines dating from the late 19th century and has been explored for large IOCG style deposits since the mid-1990s. The region was originally identified using early era airborne magnetic data commissioned by the United Nations Development Program (UNDP). This data was followed up by ground reconnaissance sampling and mapping. A relationship between copper and gold anomalism and the Hook intrusive suite was identified and an IOCG style of mineralisation postulated.

During the 1990s Billiton conducted soil geochemical surveys as well as geophysical surveys over PLLS 39 and drilled nine targets identified from the soil geochemical and geophysical work. None of these drill holes however intersected the interpreted geophysical targets due to the unexpected depth of weathering in the area. In 2000-2001, Billiton drilled another two drill holes. A summary of this work is presented in Table 6.1.

In the period 2001 to 2003/04, Billiton temporarily suspended field operations in Zambia.

In 2004, a joint venture (JV) agreement was signed between BHPB and AIM Resources Limited (subsequently Blackthorn), allowing Blackthorn to earn 70% interest in the Project by sole funding the initial three phases of exploration. BHPB subsequently withdrew from direct involvement in the project in 2011.

**Table 6.1 Pre-2004 Exploration on the Mumbwa Tenements**

| Date | Company | Activity | Parameters | Results | Comments |
|---|---|---|---|---|---|
| 1993-1994 | Billiton | Area selection for licence application based on desktop studies and digitized aeromagnetic data | | | |
| 1995 | Billiton | UNDP airborne INPUT data digitized and reinterpreted. Targets selected for follow-up work | | Integrated interpretation produced 40 potential target areas. Targets ranked with priority in the western area (abundant oxodized post-tectonic A-type intrusives with significant Cu and Fe occurrences | Target: Olympic Dam/Ernest Henry-type Fe-Cu-Au deposits using magnetic anomalies, EM conductors, favourable geology, past exploration results |
| 1996 | Billiton | Regional soil geochemical survey over EM and magnetic anomalies. Sugar Loaf and Lou-Lou deposits covered. | 500m spaced lines | Cu-Au anomaly in 3 distinct NE-trending zones, coincident with geophysical anomalies | |
| | | Ground magnetic survey over same grid | | | |
| 1997 | Billiton | Time-domain induced polarization (TDIP) survey conducted over the Cu-Au soil anomaly | Dipole-dipole survey, 100m dipole | Large number of IP anomalies obtained | |
| | | Controlled Source Magnetotelluric (CSAMT) survey conducted over the Cu-Au soil anomlay | | | CSAMT will provide better resolution and will map resistorsand conductors |
| 1998 | Billiton | Detailed soil geochemistry | 39 lines for 120km; 50m sample spacing; 200m line spacing | 27 targets defined | To improved definition of previously defined Cu-Au soil anomaly |
| | | TDIP survey | Dipole-dipole survey, 100m dipole; 6 selected lines | | |
| | | CSAMT survey | 50m stations; 35 lines | | |
| | | Ground magnetic survey | 39 lines | | |
| | | 4 DDH and 5 RC holes drilled to test soil, IP and CSAMT anomalies | Total of 1867.3m | Intersections of hematite breccia with subeconomic Cu and Au grades; extensive kaolinite and sericite alteration. | Mineralisation intersected in 8 of 9 drillholes over a strike length of approximately 6km |
| | | | | 3 targets advanced to prospect status | |
| 2000-2001 | Billiton | Reappraisal of geophysics, geochemistry and geology | | | |
| | | 2 DDHs (KD6, KD8) drilled to test hypogene mineralization below zone of oxidation | KD6 - 610.85m; KD8 - 503.83m | | |
| 2001 | Billiton | Project put on hold | | | |
| End-2004 | BHP Billiton/Aim Resources | Project joint ventured with Aim Resources | | | |

There have been seven discrete phases of exploration / drilling since 2004 (refer Section 10).

At the conclusion of Phase 4 a maiden Mineral Resource estimate was released in October 2009.

At the conclusion of Phase 5 The MSA Group conducted an updated Mineral Resource Estimate, in accordance with the guidelines of the 2004 JORC code, which was reported in June 2012. This was integrated into a Scoping Study, the results of which were released in September 2012.

Based on the positive outcome of the Scoping Study, Blackthorn committed to a further drilling programme, Phase 6, and a Pre-feasibility Study (PFS) issued in September 2013.

Phase 7 drilling comprised diamond core holes only and ran between July 2013 and November 2013 for a total of 9,533 m (including 1,302 m of drilling outside of the main Kitumba resource). An updated Mineral Resource Estimate was completed by MSA in December 2013 (MSA report submitted February 2014) and incorporated into an Optimised Pre-feasibility Study (OPFS) completed in May 2014, which forms the basis for this Technical Report.

# 7.0 GEOLOGICAL SETTING AND MINERALISATION

## 7.1 Regional Geology

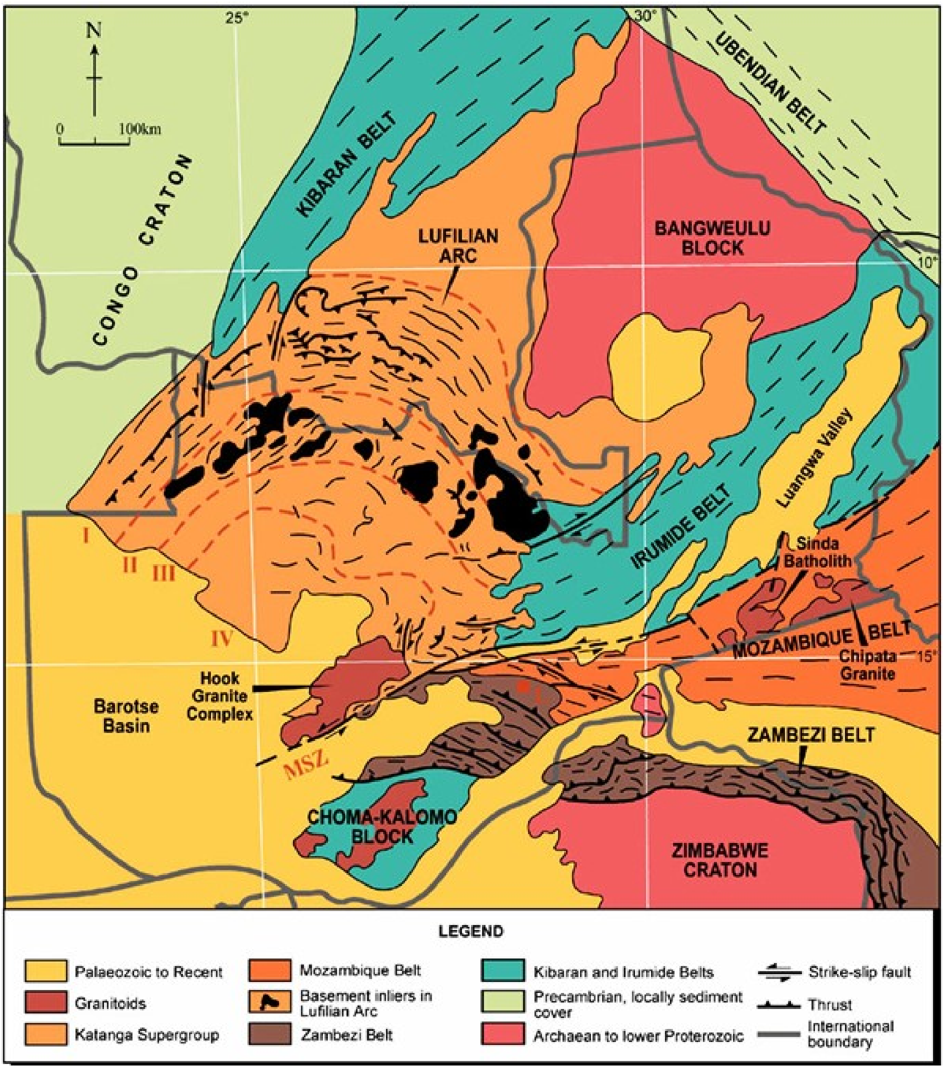
The Mumbwa licence area lies within the extreme southern portion of the Neoproterozoic Lufilian Arc. The Lufilian Arc is a large arcuate fold and thrust belt covering northwestern Zambia, the southern Democratic Republic of Congo and eastern Angola. This belt is separated from the Zambezi Belt to the south by the Mwembeshi Shear Zone (MSZ), a prominent crustal-scale east-northeast trending shear zone extending across Central Zambia. To the northwest the Lufilian Arc is flanked by the 1.4 - 1.0 Ga Kibaran Belt and to the southeast by the Mesoproterozoic Irumide Belt (Figure 7.1).

The Lufilian Arc comprises rocks of the Neoproterozoic metasedimentary Katanga Sequence intercalated with Mesoproterozoic basement inliers. The latter are exposed in a broad arc centred on the Central African Copperbelt.

Three major deformation phases are recognised within the Lufilian Arc. The first phase (D1) represents large-scale folding and thrusting throughout the Lufilian Arc. These folds have a northwest-southeast strike and are predominately north-eastward verging in the eastern part of the Lufilian Arc, strike northeast-southwest with a north-westward vergence in the western part and strike east-west in the central and northernmost part. D2 is related to east-northeast - west-southwest strike-slip faulting along the Mwembeshi Shear Zone (MSZ). Several other dislocation zones with similar trends are related to this deformation event. D3 is characterised by late transverse folding in the form of large open and upright folds trending north-northeast – south-southwest to east-northeast – west-southwest and north-south and east-west trending faults.

Most of the geological work on the Lufilian Arc has been undertaken on the northern areas centred on the Copperbelt region.

**Figure 7.1 Simplified Geology of Zambia (Modified from Ministry of Mines and Mining Development, 1999)**




## 7.2 Geology of the Mumbwa Licence Area

The geology of the Mumbwa area is described by Cikin and Drysdall (1971). The region is dominated by metasedimentary rocks of the upper units of the Neoproterozoic Katanga Sequence. These rocks are intruded by the large syn- to post-tectonic 566-533 Ma Hook Granitoid Suite and by younger post-tectonic syenites, diorites, porphyry granites, granites, diorites and gabbros. The east-northeast trending MSZ runs along the southern margin of the Hook Granitoid Suite.

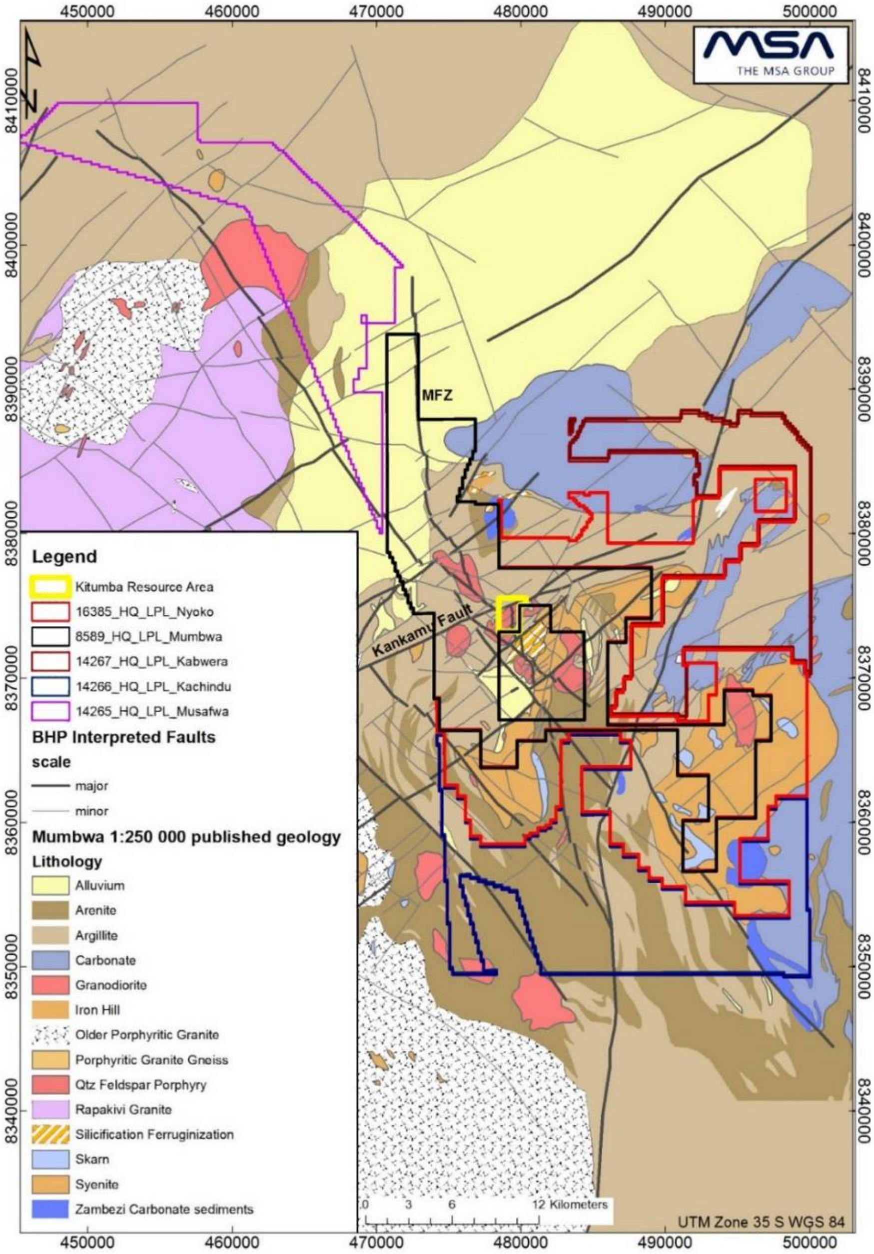
The geology of the Mumbwa licence area is shown in Figure 7.2. The northern part of the licence is underlain by metasedimentary rocks of the middle to lower Kundulungu Group of the upper Katanga Sequence. These comprise carbonates and calc-arenites interlayered with shales and siltstones. The western and southwestern part of the licence area is dominated by intrusive granitoids of the Hook Granitoid Suite. Both the sediments and the Hook granitoids are intruded by late to post-tectonic A-type syenites, quartz syenites, quartz-feldspar-porphyry granites, granites, granodiorites, diorites and gabbros.

Sedimentary rocks of the Karoo Sequence comprise a 200 m to 300 m thick succession of siltstones and mudstones developed within a northeast trending graben in the northern part of the licence. Large areas in the northwestern part of the licence area within the Kafue River flood plain are covered by Quaternary sediments consisting of alluvium and eluvium.

The Mumbwa licence straddles a regional-scale iron-oxide alteration system which is developed along a 26 km long north-northwest to south-southeast trending structural corridor referred to as the Mumbwa Fault Zone (MFZ). A structural interpretation of FALCON® magnetic and density data carried out by BHPB is shown in Figure 7.2. Apart from the MFZ, the region is dominated by a northwest and northeast trending conjugate fault set which offsets the MFZ in a number of places.

The geology, structure, alteration and mineralisation centred along the MFZ are described in detail in an internal BHPB report by Hayward (2008). Two contiguous magmatic hydrothermal systems are developed along the MFZ corridor, centred on strong magnetic anomalies representing intense magnetite-amphibole-apatite alteration spatially associated with syenite porphyry intrusions. The southern Kitumba system is separated from the northern Mutoya system by the Kankamu Fault. Both magmatic systems are zoned from deeper level magnetite-dominated alteration in the south to higher level hematite-rich breccias in the north. The Mutoya system is much larger and extends in a north-south direction for 19 km along the MFZ, being defined by a prominent linear density anomaly and coincident magnetic anomaly in the south. The northern portion of Mutoya is overlain by up to 200 m of Karoo sediments.

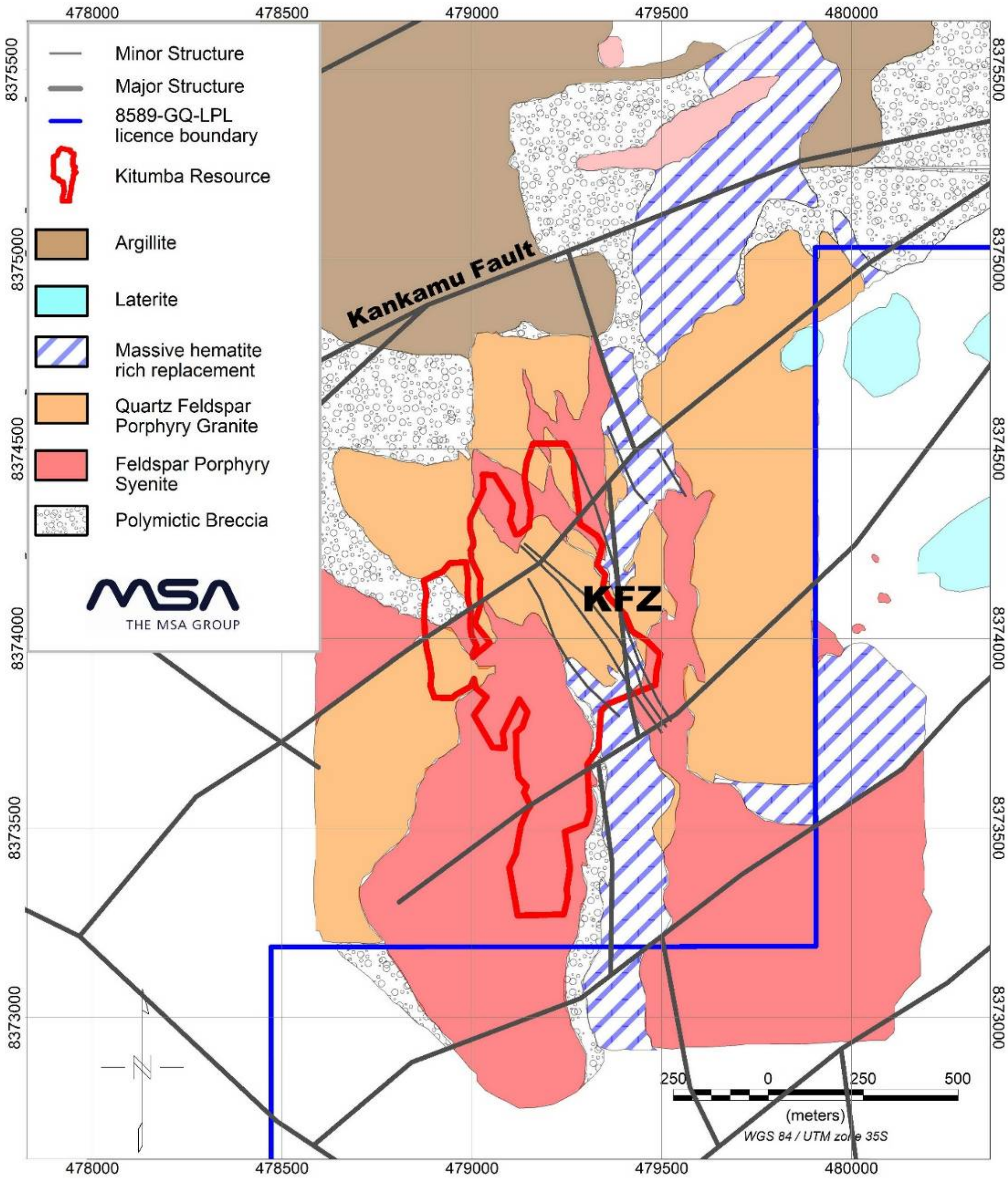
**Figure 7.2 Geology and Structural Interpretation of the Mumbwa Licence Area**




## 7.3 Geology of the Kitumba Deposit

The Kitumba mineral system is developed over a strike length of approximately 3 km and has been the focus of much of the Phases 2 to 7 exploration and resource drilling campaigns. The geology of the area was remapped by BHPB in 2009 and is shown in Figure 7.3.

**Figure 7.3 Geology of the Kitumba Area**




The Kitumba IOCG deposit is hosted within a hematite-dominated breccia system which is developed along the Kitumba Fault Zone (KFZ) and which outcrops as a prominent north-south trending ridge forming part of the Kitumba Hills. The KFZ is considered to represent a local structure within the broader MFZ. Three principal rock-type associations are recognised at Kitumba. Kundulungu Group metasediments are intruded by quartz-feldspar porphyry granitoids which are in turn intruded by a feldspar porphyry diorite / syenite complex. In addition, breccias occurring as distinct lithologies, and pervasive iron oxide replacement (IORE) of various protoliths are a common feature. Typical examples of these main rock types are shown in Figure 7.4. The geometry of this system is considered to be sub vertical and complex, arising from several phases of intrusion commonly in the form of dyke swarms. This makes correlating geology between drill holes difficult.

**Figure 7.4 Principal Lithologies at Kitumba**

A) Steeply dipping and ripple cross laminated metasiltsone, with incipient brecciation, hematite alteration and pyrite mineralisation

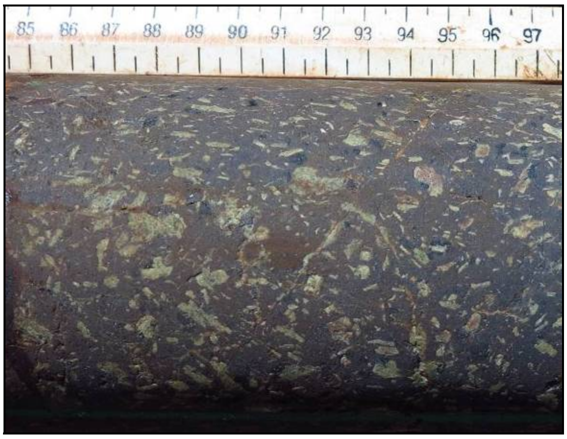
B) Thinly bedded upward fining metasediments

C) Quartz porphyry granite

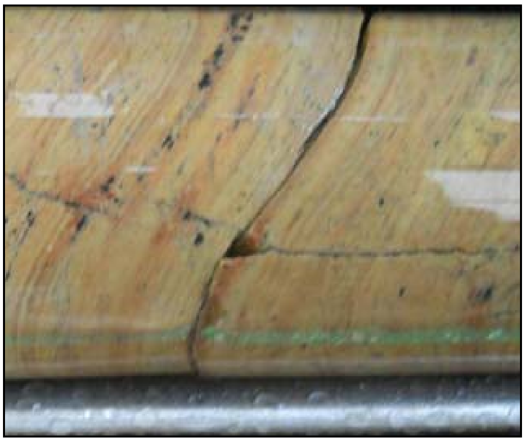
D) Feldspar porphyry diorite/syenite intruding quartz porphyry granite (note xenoliths)


E) Feldspar porphyry syenite

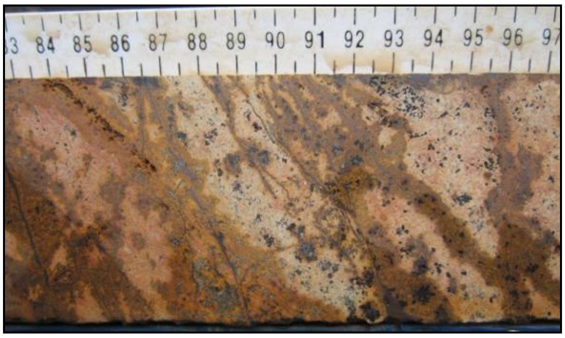
F) Fragmental polymictic breccia




G) Feldspar porphyry diorite with sericite replacement of plagioclase phenocrysts (KITDD_022, 80 m)



H) Laminated siltstone (KITDD_020, 320 m




I) Syenite with hematite veining

J) IORE (Iron Oxide Replaced)

The Kundulungu metasediments comprise thinly bedded calcareous siltstones and argillites with subordinate arenaceous partings (Figure 7.4 B and H). These become more prevalent towards the north and west, away from the core of the Kitumba magmatic system, where they dip steeply to the south and east. Within the central parts, metasediments occur mainly as breccia clasts at various scales, where they are commonly strongly altered. The metasediments also occur in the western part of the Kitumba system with a distinct laminated texture (Figure 7.4 H). The porphyry-textured granite is characterised by rounded quartz phenocrysts and to a lesser extent subequant feldspar phenocrysts, within a grey to pink groundmass determined by the dominant alteration present (Figure 7.4 C).

Both the metasediments and porphyry granitoids are extensively intruded by feldspar porphyry diorite / syenite (protolith is considered to be a diorite, which due to extensive potassic alteration has the appearance of a syenite and was consistency logged as syenite during the course of the various drilling campaigns). Based on drilling data, syenite occurs both as a massive stock and as dykes and is generally extensively altered by K-feldspar, giving the rock a characteristic red colouration (Figure 7.4 D and E) or altered by sericite, giving the rock a distinctive olive grey colouration. Sericite also replaces what are thought to be plagioclase phenocrysts. Fine grained narrow micro-syenite dykes are also observed.

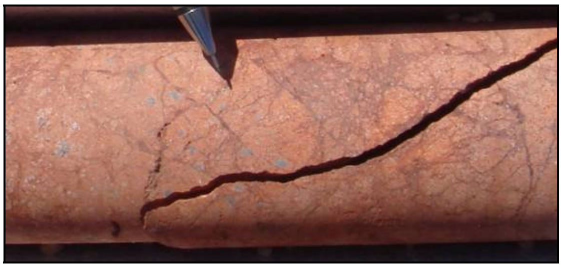
On the eastern side of the KFZ, KITDD_012 and KITDD_022 intersected a grey porphyritic rock containing no K-feldspar alteration and was thus termed a feldspar porphyry diorite (Figure 7.4 G).

Widespread brecciation is observed throughout the Kitumba area, ranging from incipient crackle breccias through jigsaw, mosaic, rubble, clast supported, matrix-supported, disaggregation and polyphase breccias to corrosive transported (fluidised) matrix and clast-supported breccias (Figure 7.5). Transported breccias are described as a lithology type as opposed to a breccia texture only. The central north-south trending ridge along the KFZ has been mapped as a 100 m to 400 m wide zone of brecciation and hematite replacement.

The KFZ is crosscut by a series of northeast trending faults, with associated dextral displacements. The central part of the deposit is crosscut by a zone of (later?) north-northwest to northwest trending intense faulting and fracturing (Figure 7.3). This structural framework resulted in development of secondary permeability which appears to have played a major role in remobilisation and supergene concentration of copper and gold mineralisation.

A classic IOCG type alteration system is observed at Kitumba. The system is zoned from north to south, with deeper level magnetite dominated alteration south of Kitumba to the Sugar Loaf deposit (3.8 km southwest of Kitumba) and higher level hematite dominated alteration over Kitumba itself. These correspond to respective magnetic highs and lows in the FALCON® data.

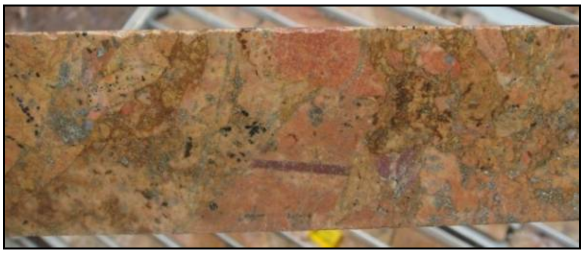
**Figure 7.5 Examples of Principal Breccia Types at Kitumba from Selected Drillholes**



A) Crackle breccia in quartz porphyry granite (S36-018)

B) Mosaic breccia in syenite (S36-001, 32 m)

C) Transported hematite breccia



D) Possible polyphase breccia with pyrite mineralisation (S36-017)


E) Brecciated hematite matrix supported metasediments (S36-015)

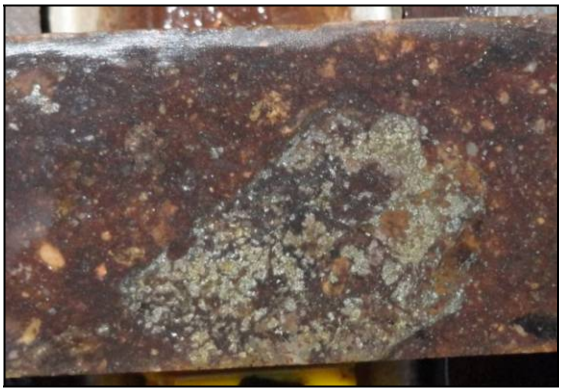

F) Matrix supported breccia in metasediments (KD8)


G) Matrix supported rubble breccia (S1-001)

H) Polymictic matrix supported rubble/fluidised breccia (S1-001)

I) Fluidised flow breccia (S36-018, 97 m)

J) Mosaic breccia in altered quartz porphyry granite (S36-015)

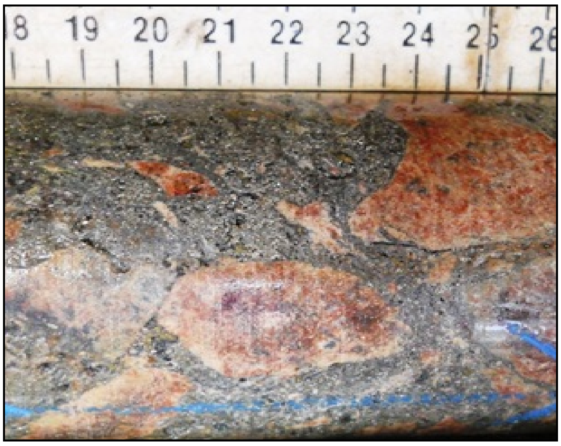


K) Pyrite mineralised breccia clast (S36-001)

L) Carbonate-matrix breccia clast within iron oxide matrix breccia – evidence for polyphase brecciation (S36-018)

M) Mineralized breccia with iron oxide matrix and K-feldspar and sericite altered clasts. (KITDD_009, 413 m)

N) Sub-horizontal flow breccia with pervasive carbonate alteration and late siderite veins. Brecciation indicates the proximity to the KFZ (KITDD_005, 603 m)



O) Brecciated quartz feldspar porphyry granite with pyrite and chalcopyrite matrix (KITDD_011-2, 463 m)

P) Polymictic matrix supported rubble/fluidised breccia with rounded clasts (KITDD_001)

The dominant alteration facies are described below and examples shown in Figure 7.6:

- Magnetite-amphibole-apatite-carbonate alteration

  Magnetite-amphibole-apatite-carbonate alteration is dominant in the area south of Kitumba and in the Mutoya area north of the Kankamu Fault (Figure 7.6 J). These areas are associated with magnetic highs and represent uplift and tilting along northeast trending faults.

- K-feldspar alteration

  K-feldspar alteration is widespread and ranges from selective replacement of feldspar phenocrysts of porphyry granite and diorite/syenite, to clasts within earlier breccias, to pervasive replacement of all three main rock types (Figure 7.6 A). This alteration facies is most intensely developed in syenite and porphyritic syenite and is often associated with copper mineralisation, especially in Kitumba. The potassic alteration zone appears to be bound to the east by the KFZ which suggests that displacement along the KFZ placed potassic altered and non-potassic altered fault blocks adjacent to each other. This suggests that potassic alteration occurred early in the life of the KFZ and was not active in the later phases of reactivation when hematite alteration is prevalent. The potassic alteration zone is open to the west of Kitumba, but has been noted to decrease in intensity to the west.

- Sericite alteration

Sericite alteration in Kitumba is poorly understood. It typically overprints K-feldspar alteration (Figure 7.6 E) and is more prevalent in distal portions of the resource area, potentially extending further west than the potassic alteration event. Sericite alteration is also not documented immediately to the east of the MFZ. Sericite alteration appears to be more intense in the distal portions of the Kitumba area, but it has been noted to replace plagioclase phenocrysts in the FPSY lithology even in the centre of the resource area (KITDD_005). Controls on sericite alteration and the possible link to sulphide mineralisation are not understood.

- Carbonate alteration

Carbonate alteration in Kitumba is common and two main varieties have been identified. Late siderite-ankerite veining occurs commonly through the resource area and crosscuts all other alteration zones and veins including hematite, K-feldspar and sericite alteration (Figure 7.6 O and Figure 7.6 N). The carbonate veining is usually in the form of siderite within the Kitumba deposit proximal to the KFZ, and more calcium rich (calcite and ankerite) distally. Siderite-ankerite veins are commonly found with pyrite and chalcopyrite blebs and disseminations.

The second form of carbonate alteration is pervasive siderite alteration. It is typically difficult to identify in fresh core because it reacts weakly with dilute hydrochloric acid. A characteristic tarnish develops after the core is exposed to air for a few months which assists in identification. Pervasive siderite alteration is abundant throughout the hypogene zone and often occurs with potassic and sericite alteration. Carbonate alteration is absent in intensely hematite altered zones and in kaolinite altered zones. The relative timing of the pervasive carbonate alteration is poorly understood, however its importance cannot be understated as its presence is a fundamental control on the hypogene - supergene boundary.

- Hematite alteration

Hematite alteration is widespread at Kitumba but is most intense proximal to the KFZ where it is associated with large-scale breccia development and the development of stockwork veining. Hematite replacement varies from partial to pervasive and occurs as various shades of red-brown hematite (Figure 7.6 H and I), as well as primary grey specular hematite (Figure 7.6 P). Four separate forms of hematite alteration occur, each with a distinctly different genesis:

- The red-brown variety is regarded as secondary iron alteration and results from interaction with downwards percolating iron bearing meteoric fluids associated with weathering. Secondary hematite occurs as veins, sometimes with distinctive haloes and also as a pervasive hematite wash. Hematite alteration is observed to overprint K-feldspar, sericite and pervasive carbonate alteration. Secondary hematite alteration is an important indicator of weathering and has a strong control on supergene mineralisation.

- Specular hematite alteration is considered primary and results from interaction with hotter fluids at depth controlled by fluid movement on the KFS. It occurs as veins and blebs.

- Hematite associated with quartz veins (Figure 7.6 Q) occurs occasionally in the Kitumba deposit and is not considered important in mineralising processes.

- Steely hematite occurs in the shallow portion of holes and is believed to result from regolith processes relating to the weathering of primary hematite as well as induration and enrichment in the regolith profile.

The alteration zoning in Kitumba is complex with several different phases overprinting each other or occurring concurrently. A schematic interpretation of the extents of the important alteration zones are shown in Figure 7.7. The interpretation is based on geological modelling of core logs and geochemistry.

**Figure 7.6 Examples of Principal Alteration Types at Kitumba from Selected Drillholes**

A) Pervasive K-feldspar alteration of syenite (KITDD_009)


B) Potassic alteration partially overprinted by hematite alteration (S36-001)

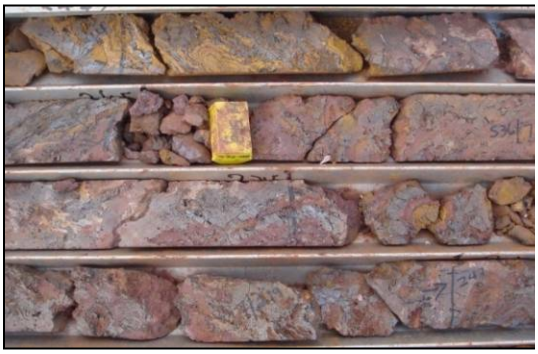
C) Polyphase hematite alteration of fluidised breccia in S36-015, with disseminated native copper

D) Silica alteration overprinting pervasive hematite replacement (KD3)

E) K-feldspar alteration overprinted by sericite and subsequently carbonate veining (S3-001)

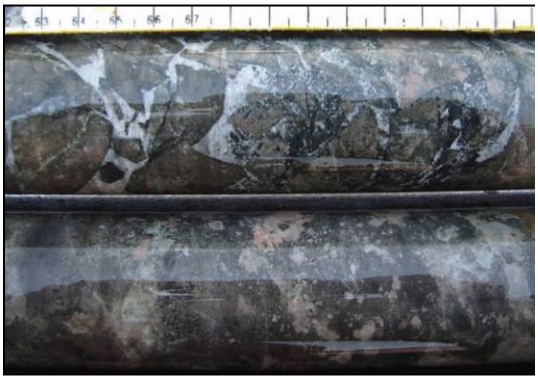
F) Potassic altered quartz porphyry granite cut by carbonate-pyrite vein (S3-001)

G) Pervasive carbonate replacement of rubble breccia



H) Pervasive hematite replacement of metasediments (S36-007)

I) Hematite replacement fronts associated with stockwork veining in syenite



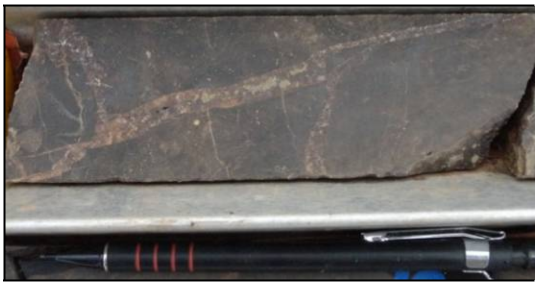
J) Magnetite-amphibole-apatite alteration in metasediments (S30-001)

K) Hematite alteration fronts emanating from hematite-filled fractures (S36-001, 20 m)

L) Contact between oxidised zone and carbonate alteration (S36-001, 467 m)

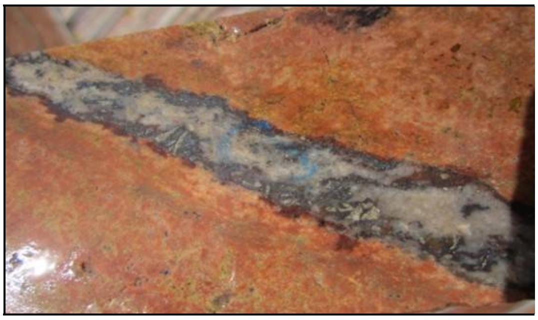
M) Carbonate-altered (bleached by acid) and oxidised quartz porphyry granite (S36-018)

N) Pervasively iron oxide altered metasedimentary breccia clasts (laminated), Carbonate matrix breccia overprinted by iron oxide alteration in the top row (S36-018)



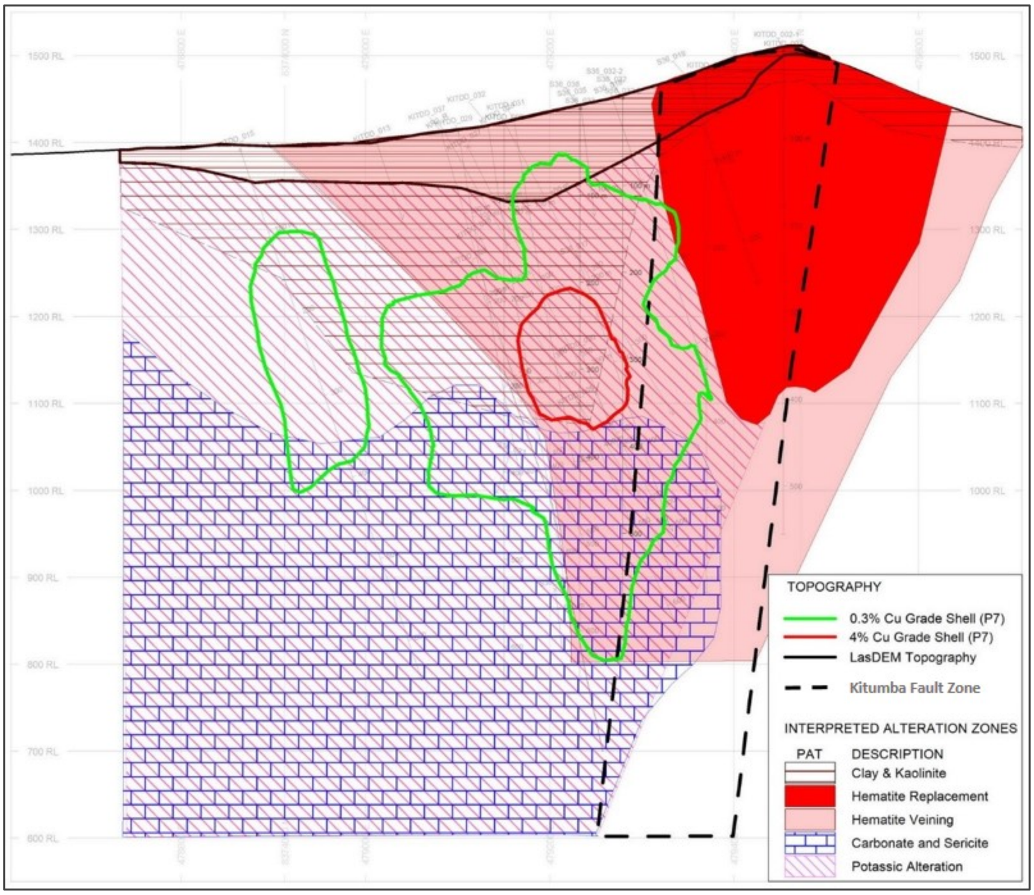
O) Pervasive carbonate altered syenite crosscut by carbonate-pyrite veining (S36-018)

P) Specular hematite vein with a large secondary hematite halo. (KITDD_009, 550 m)



Q) Hematite - quartz vein cutting through K-feldspar and sericite altered syenite (KITDD_009, 483 m). Mineralisation is found only in the hematite vein

**Figure 7.7 West-East Schematic Section through the Kitumba Deposit showing Alteration Zoning**




## 7.4 Mineralisation

### 7.4.1 District-Scale IOCG Mineralisation

The Mumbwa district represents an IOCG province that is related to voluminous anorogenic (A-type), alkali and granitoid magmatism and tectonic activity that took place from 570-500 Ma along a Pan-African transform plate boundary that separated the Congo and Kalahari Cratons (Pelly, 2001). IOCG systems associated with the Hook Granitoid Suite and related intrusions in the Mumbwa district display many of the typical characteristics of IOCG systems. Numerous copper-gold and iron oxide occurrences are known in the district and are noted on the Geological and Mineral Occurrence Map of Zambia.

Diorite / syenite intrusions into Kundulungu metasediments, iron oxide alteration and brecciation are also known from the Kasempa region approximately 150 km to the north of Kitumba.

### 7.4.2 Deposit-Scale IOCG Mineralisation

The Kitumba deposit and its geological setting display similar characteristics to other IOCG deposits, including Olympic Dam. Typical characteristics include:

- Copper-gold mineralisation accompanied by large-scale partial to pervasive iron alteration and brecciation.
- Vertically oriented, structurally controlled hematite-rich breccia zones containing pyrite and chalcopyrite.
- Mineralisation and brecciation spatially associated with major structural intersections.
- A spatial and temporal association with the multi-phase Hook Granitoid Suite, and extensive potassic alteration.

According to Hitzman and Broughton (2003), intrusive and volcanic rocks within these systems have undergone pre-mineralisation potassic alteration characterized by the formation of potassium feldspar. Potassically altered rocks are then cut by magnetite, which is in turn replaced by hematite during a hydrolytic (sericite-chlorite) alteration event. Sulphidation is the final event resulting in the precipitation of pyrite and then chalcopyrite.

Copper mineralisation at Kitumba comprises a simple hypogene sulphide assemblage that has been extensively overprinted by a complex largely redistributed supergene oxide assemblage. Kitumba represents a deeply weathered IOCG system with weathering and oxidation extending to 500 m depth in places. Deep weathering is particularly pronounced in the vicinity of the KFZ and zones of high fracture intensity, where leaching of sulphides has generated acid, resulting in a porous and vuggy core, in places iron is preserved or enriched resulting in steely hematite mega mottles and liesegang structures surrounded by weathered saprolite in the leached zone. Hayward (2008) considers this the reason for the unusually low density of these iron-rich rocks and explains why Kitumba is not mapped as a density anomaly in the FALCON® data. Oxidation of sulphides is noted at significant depths in some holes.

Hypogene mineralisation at Kitumba mainly occurs as disseminated pyrite and subordinate chalcopyrite, associated with stockwork veining and brecciation. Semi-massive concentrations of pyrite ± chalcopyrite are observed in places and are most concentrated beneath the richest supergene mineralisation adjacent to the KFZ. Primary sulphides are largely preserved within iron carbonate altered zones, sometimes relatively shallow, where carbonate has acted as a buffer preserving hypogene assemblages.

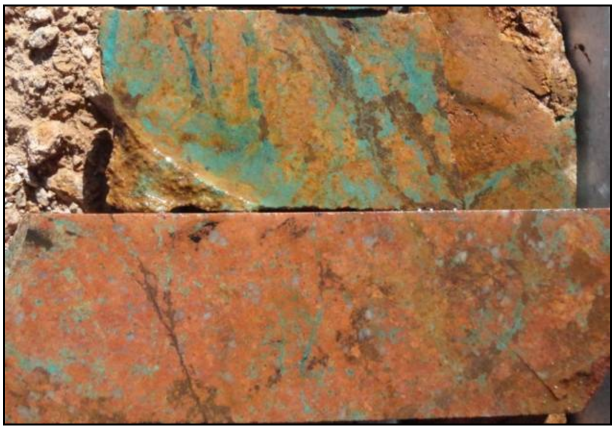
A significant proportion of the mineralisation occurs in the form of secondary copper minerals within the supergene zone. Supergene enriched material is concentrated from 200 m below surface where it occurs as chalcocite ± malachite, pseudomalachite, chalcosiderite, cuprite, native copper, and copper wad (within limonite). The distribution of secondary copper minerals is related to remobilisation of copper; with secondary copper minerals commonly occurring along fractures and as linings in cavities.

The zone of north-northwest to northwest trending intense faulting and fracturing in the south central part of the deposit (Figure 7.3) has resulted in deep oxidation of the host rocks and remobilisation and re-concentration of copper mineralisation particularly along fractures and cavities, forming a high grade core to the Kitumba deposit. Particularly high grade supergene mineralisation is developed at the base of such zones, immediately above the contact with pervasively carbonate altered host rock.

The upper part of the system is variably leached, and is typically depleted in copper in the upper 200 m. The Phase 5 Mineral Resource estimate defined zones of potential shallow lower grade copper ± gold mineralisation in places. These were followed up in the start of Phase 6 and no shallow copper mineralisation was found near the MFZ, however shallow gold enrichment was found in drill hole KITDD_010.

Examples of the various mineralisation styles are shown in Figure 7.8.

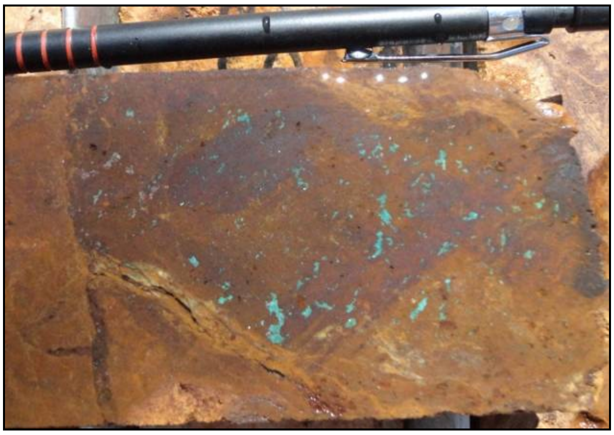
**Figure 7.8 Examples of Mineralisation Types at Kitumba from Selected Drillholes**



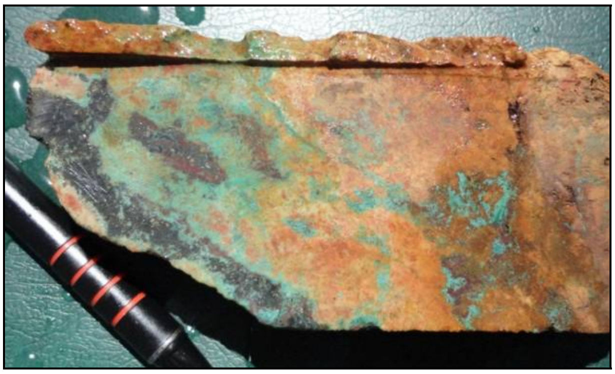
Malachite and pseudomalachite along fractures in quartz porphyry granite (S36-001; 126 m)

Pseudomalachite in brecciated syenite (S36-017)

Pseudomalachite fringing hematite alteration in syenite (S36-001)

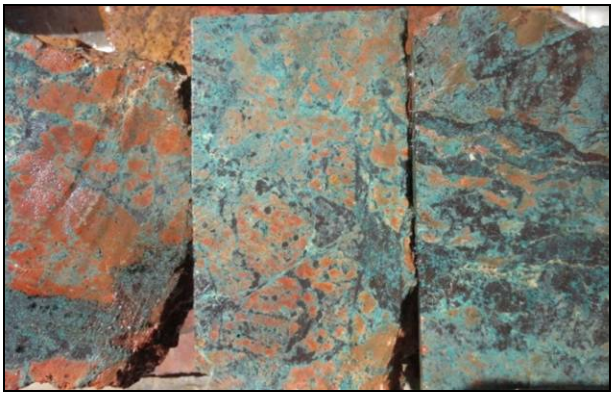


Pseudomalachite in breccia clast (S36-001; 207 m)

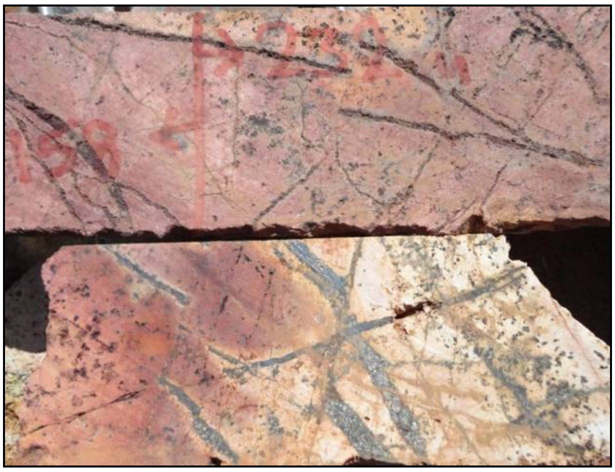


Malachite + pseudomalachite cored by chalcocite ± pyrite
(S36-017 306.2 m)

Malachite + pseudomalachite cored by chalcocite
S36-038 220.8 m)



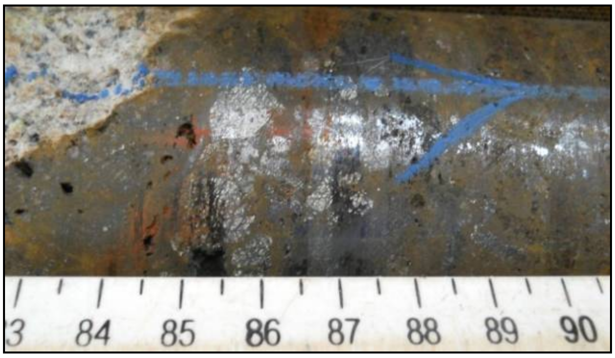
Malachite + pseudomalachite cored by chalcocite (S36-038 224.5 m)



Fracture-hosted chalcocite pyrite in K-feldspar altered syenite
(S36-038, 232 m)

Malachite, pseudomalachite and chalcocite in weathered brecciated syenite (S36-033, 206 m)

Remnant pyrite surrounded by digenite chalcocite and limonite, (KITDD_005, 310 m)

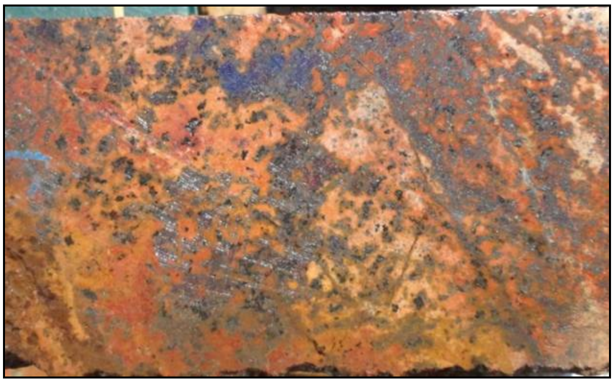



Semi-massive chalcocite in altered syenite within high-grade supergene zone (KITDD_019, 354.35 m)

Chalcocite cemented breccia within high-grade supergene zone (S36-038, 236 m)

Chalcocite developed mainly along fractures in syenite (S36-033, 222.5 m)

Chalcocite and native copper within syenite (KITDD_014, 395 m)



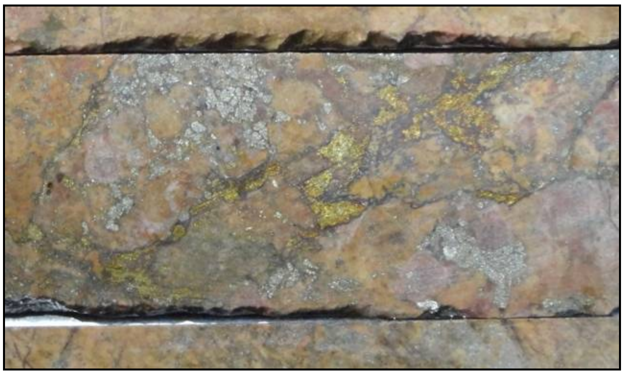
Fracture-hosted + disseminated chalcocite + bornite (S36-038, 236 m)

Native copper and calcite in vugs, hematite breccia (S36-015)


Fracture-hosted cuprite + malachite in altered syenite
(S36-036, 298 m)

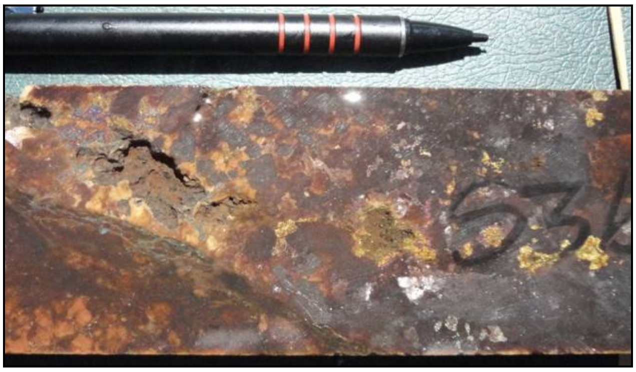

Secondary chalcocite + malachite in oxidised syenite in contact with hypogene pyrite chalcopyrite in carbonate altered syenite
(S36-033, ~408 m)

Fracture-hosted pyrite ± chalcopyrite in carbonate-altered syenite
(S36-033, 384 m)

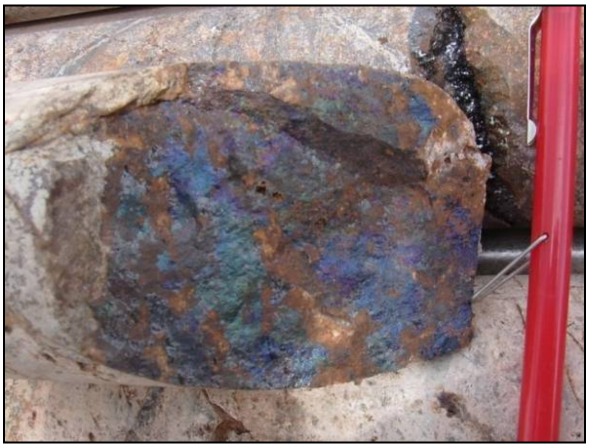


Fracture-hosted pyrite and chalcopyrite in altered syenite
(S36-033,413 m)


Semi-massive chalcopyrite in altered syenite (KITDD_008, 520 m)

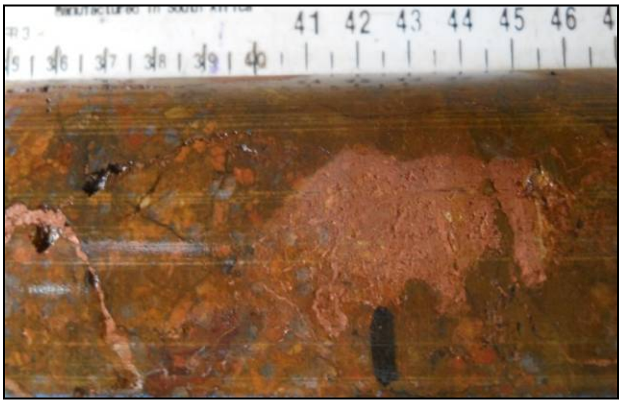


Coarse carbonate alteration with chalcopyrite rimmed by bornite
(S36-017, 430.5 m)



Bornite on fracture surface (S4-001)

Chalcopyrite-sericite veining in quartz porphyry granite, crosscut by carbonate vein




Native copper in quartz porphyry granite (KITDD_018, 308 m)


Pyrite and earthy chalcocite vein (KITDD_016, 284 m)

# 8.0 DEPOSIT TYPES

Kitumba is considered to be an iron oxide – copper - gold (IOCG) type deposit.

Such deposits cover a diversity of types and range from Kiruna-type monometallic (Fe ± P) to Olympic Dam type polymetallic (Fe ± Cu ± Au ± U ± REE) systems. IOCG-type deposits have been reviewed by, amongst others, Sillitoe (2003), Hitzman (2000), Lefebure (1995) and Corriveau (2007), who describe the following characteristic features:

- IOCG deposits comprise a broad and ill-defined clan of mineralisation styles, which are grouped together chiefly because they contain hydrothermal magnetite and/or specular hematite as major accompaniments to chalcopyrite ± bornite. Apart from copper and by-product gold appreciable amounts of Co, U, REE, Mo, Zn, Ag, Nb and P may also be present.

- Deposits vary between magnetite-apatite deposits with actinolite or pyroxene (Kiruna type) and hematite-magnetite deposits with varying amounts of copper sulphides, Au, Ag, uranium minerals and REE (Olympic Dam type). Typically, these systems are characterized by >20% iron oxides. Iron-rich zones, breccias and alteration halos associated with IOCG systems can reach hundreds of metres in width and many kilometres in length.

- Some of the deposits (typically hematite-rich) are characterised by breccias at all scales, with iron oxide and host rock fragments which grade from weakly fractured host rock to matrix-supported breccia (sometimes heterolithic) to zones of 100% iron oxide.

- Deposits form at shallow to mid-crustal levels in regionally oxidised environments and are typically associated with extensional environments such as intracratonic and intra-arc rifts and continental magmatic arcs.

- Deposits are localized along high- to low-angle faults which are generally splays off major, crustal-scale faults. Structural control can vary from the intersection of highly permeable units with fault zones, dilational jogs, duplexes, splays on faults and shears, folding or complex intercalation of high and low permeability units all influencing fluid flow regimes and ultimately the position of alteration zones, breccias and/or ore deposition.

- IOCG systems are generally sulphur deficient magnetite and/or hematite bodies of hydrothermal origin. The morphology of IOCG mineralisation varies significantly and includes breccia zones, veins and irregular bodies that may occur as stratiform, stratabound or discordant deposits and disseminations to massive lenses, hosted by continental sediments and/or volcanics and/or intrusive rocks.

- The vast majority of IOCG deposits are spatially and temporally related to a significant magmatic event although the spatial relation is not necessarily a direct one with a specific intrusion or a specific type of magma. Reference is often made of the presence of A-type or I-type granites on a regional scale.

- Host rocks to IOCG deposits are generally intensely altered with alteration mineralogy largely dependent on the host rock type. Zoned hydrothermal alteration is characteristic of many IOCG deposits with alteration patterns depending on depth of emplacement and host rock chemistry (Lobo-Guerrero, 2003).

- A general order of hydrothermal alteration and mineralisation is: i) country rock albitization; ii) high temperature Fe ± K ± Ca alteration (biotite - garnet - K-feldspar - amphibolite - clinopyroxene - magnetite); and, iii) mineralisation and low temperature alteration (sulphides ± magnetite ± hematite - carbonate ± sericite/chlorite) with possible Cu and/or Au. Corriveau (2007) also notes that the main ore stage is associated with potassium-iron alteration zones in which either sericite or K-feldspar prevail as the potassic phase.

- Diagnostic alteration patterns are typical with regional calcic-sodic alteration superimposed by focused potassic and iron oxide alteration. Zoned hydrothermal alteration is characteristic of many IOCG deposits.

- At Olympic Dam, Cu is zoned from a predominantly hematite core (minor chalcocite-bornite) to a chalcocite-bornite zone then bornite-chalcopyrite to chalcopyrite-pyrite in the outermost breccia. Uraninite and coffinite occur as fine-grained disseminations with sulphides; native gold forms fine grains disseminated in the matrix and inclusions within sulphides.

# 9.0 EXPLORATION

The Kitumba deposit is part of a much larger IOCG system which is potentially mineralized elsewhere along its 26 km north-south strike length. Evidence for this is observed at Kakozhi, 5 km to the northwest of Kitumba. In addition, significant exploration potential exists for the discovery of further IOCG-type mineralisation elsewhere along the continuation of the Mumbwa Fault Zone, particularly at the intersections of northeast trending faults, where soil geochemical anomalies have been defined in previous work.

This section describes the nature and extent of work other than drilling (covered in Item 10) and the geophysics survey (undertaken by BHPB and described in Item 9.1) undertaken by Blackthorn within the Mumbwa tenement area.

## 9.1 Mushingashi Induced Polarisation Survey

BHP Billiton (BHPB) took over management of the Joint Venture project in August 2008, and in 2010 shifted focus to the Mutoya-Mushingashi area which is characterized by a 20 km long north-south trending FALCON® density anomaly along the extension of the Mumbwa Fault Zone. Targeting a giant Olympic Dam-type system, BHPB undertook an Induced Polarisation (IP) survey to screen for potential sulphide accumulations within and peripheral to interpreted iron oxide alteration systems extending northeast of Kitumba under Karoo cover in the Mutoya-Mushingashi area. The survey was also designed to test targets derived from the original inversion modelling of FALCON® airborne gravity gradiometry and magnetic data.

The survey was acquired using a 2D pole-dipole configuration employing 200 m receiver dipoles. Up to 16 contiguous receiving dipoles were deployed per spread, equating to a line coverage of 3.2 km. Eight east-west trending lines were surveyed.

The IP survey successfully mapped the extent (>300 m) of conductive Karoo cover as well as elevated sulphide (pyrite) concentrations intersected in the Phase 4 drilling. The linear density anomaly is due to an extensive zone of manganosiderite alteration. Disseminated suphides (mainly pyrite) hosted within this and within the adjacent metasediments, together with black ‘steely’ hematite, were concluded to be the source of chargeability anomalies defined by the IP survey.

## 9.2 Remodelling of FALCON® Data

PGN Geoscience (PGN) was commissioned by Blackthorn to carry out remodelling and interpretation of the FALCON® airborne gravity gradiometry, gravity equivalent and magnetic data covering the Kitumba, Mutoya and Mushingashi areas. The results of the study were reported in 2011 (Ailleres, L. 2011a and 2011b). The FALCON® survey was flown by Fugro Airborne Surveys in 2004 and was the catalyst towards advancing IOCG exploration efforts in the Mumbwa area, which lead to the Kitumba discovery in 2007 (Christensen and Whiting, 2013).

The magnetic and equivalent gravity data were inverted, with petrophysical data from drill holes used in the inversions. Priority targets were selected on the basis of magnetic highs spatially just off gravity highs and in proximity to crosscutting northeast-southwest and northwest-southeast structures spatially associated with a jog in the regional north-south trending Mumbwa Fault Zone. Higher resolution inversions were carried out over selected target areas to better depth estimates for drill testing. The improved inversions confirmed Kakozhi as the main target outside of Kitumba.

As is the case at Kitumba, Kakozhi is marked by a significant radiometric uranium anomaly. Drill testing of the Kakozhi target is discussed in Item 10 of this report. Several other targets satisfying the same criteria are identified to the immediate north of Kakozhi and remain to be tested.

Based on remodelling of the FALCON® data, four individual IOCG prospects (Kitumba, Mutoya, Kakozhi and Mushingashi) have been identified, together comprising a giant IOCG alteration system developed over a 26 km long north-northwest to north-south trending structural corridor defined by the Mumbwa Fault Zone (Christensen and Whiting, 2013). 3D inversion of the magnetic and gravity gradient data has assisted in mapping the alteration systems within these prospects from distal high temperature magnetite alteration to proximal lower temperature hematite alteration.

## 9.3 Kakozhi Mapping and Soil Geochemical Sampling

The Kakozhi target (initially called 'Target A') was generated through remodelling and interpretation of the FALCON® magnetic, radiometric and gravity data. Kakozhi is located approximately 5 km northwest of the Kitumba deposit. In order to assist in drill testing of the modelled geophysical targets, a detailed geological mapping and soil geochemical survey was carried out at Kakozhi in early 2012.

Geological mapping was conducted concurrently with soil sampling on a 100 m x 100 m grid along east-west traverse lines over an area of 6.9 $km^2$ (Figure 9.1) Geological data were recorded onto field mapping sheets which were subsequently digitised. The Kakozhi area is dominated by two prominent hills, with a western hill comprising iron oxide replaced rock (referred to locally as IORE) and an eastern hill of quartz-feldspar porphyry granite. The geological setting is similar to that at Kitumba, with quartz-feldspar porphyry granite and subordinate microgranite and fine-grained syenite intruding Kundulungu metasedimentary rocks. Although brecciated in places the intrusives are undeformed and post-tectonic.

Kundulungu metasedimentary rocks (sandstone and siltstone) outcrop predominantly in the southern and south-western parts of the area. Minor carbonate is mapped and contains disseminated pyrite and subordinate chalcopyrite in places.

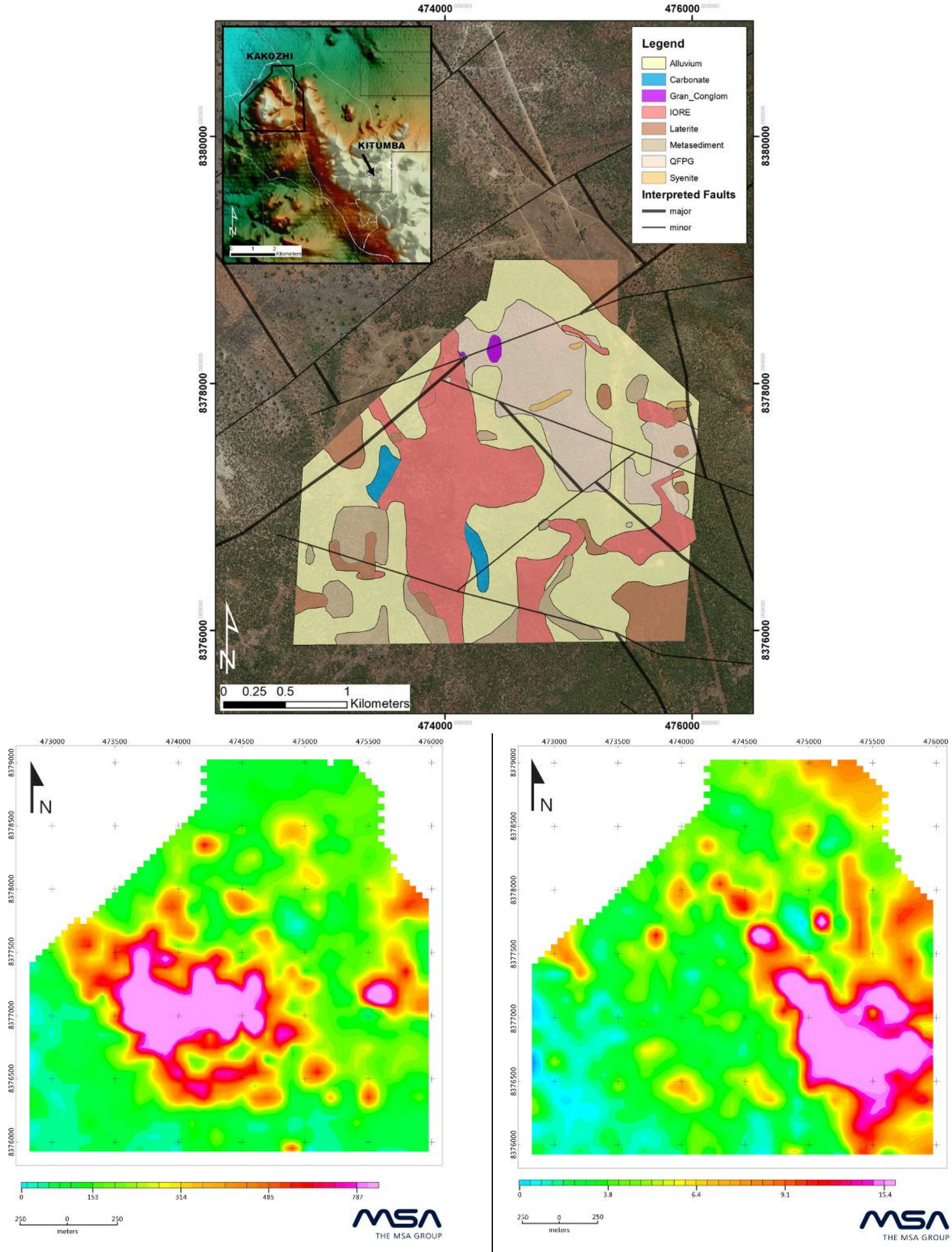
Both the intrusives and metasediments have been variably affected by iron oxide alteration in the form of specularite, hematite and magnetite veins with pervasive texturally destructive ferruginisation in places. A large part of the area is underlain by rocks pervasively altered by iron oxide (IORE). These zones are hematite-dominated in the west and magnetite-dominated in the east, and are brecciated in places. Pervasive iron alteration along a prominent north-south trending zone in the western part of the area is considered to be related to a structure.

The remainder of the mapped area is covered by alluvium and laterite pavements.

Soil geochemical sampling involved collection of a 1-1.5 kg soil sample from the B horizon at a nominal depth of 40 cm. In some cases shallower holes were dug or no sample was collected due to poorly developed or absent soil horizons characteristic of elevated and lateritic areas. Sample numbers were recorded on sample tickets stapled onto the folded plastic sample bags and on aluminium tags inserted into the sample bags. Quality control blanks and certified reference materials were inserted into the sample stream at a frequency of 1 in 20 samples. All relevant information at each sample site was recorded on a standardised soil sampling sheet and subsequently captured into the project database.

Copper soil anomalies correlate well with the IORE exposures (Figure 9.1). Copper results from lithogeochemical sampling are low, probably as a result of leaching. At Kitumba, extensive leaching has removed much of the copper from the upper 200 m, although an associated copper soil anomaly overlies the deposit. Uranium in soil at Kakozhi shows a strong northwest trend, correlating with an interpreted structure (Figure 9.1).

**Figure 9.1 Kakozhi Geology and Gridded Cu and U Soil Geochemistry**




ppm Cu

ppm U

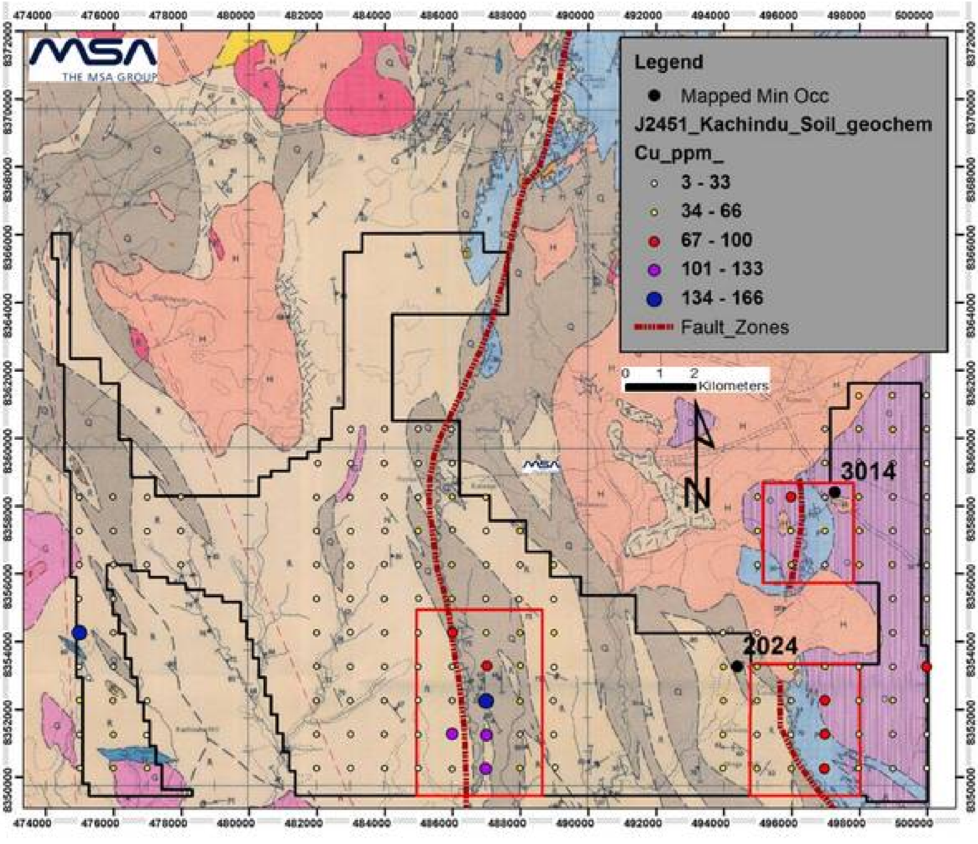
## 9.4 Regional Soil Geochemical Sampling - 2012

Regional soil geochemical sampling was carried out on the Musafwa and Kachindu licences in 2012. The sampling covered three areas of interest on Kachindu and one area of interest on Musafwa, selected on the basis of their spatial association with fault zones and/or late-stage granitoids intruding Kundulungu metasedimentary rocks.

Sampling was carried out along 1 km spaced east-west lines, with samples collected on a 200 m spacing and composited over 1 km. Samples were collected from the top of the B horizon at a nominal depth of 30 cm to 40 cm and placed in plastic sample bags. Sample tickets were stapled to the bags and the sample number also recorded on aluminium tags placed inside the bags. All relevant information at each sample site, including geological observations, was recorded on standardised field sheets.

Samples were submitted to the Intertek Genalysis laboratory in Perth for multi element analysis by Aqua Regia digest and ICP finish. Results for the IOCG pathfinder elements were low, particularly in the case of the Musafwa licence, and partly a result of compositing of samples. In the Kachindu licence, low level copper is observed in two of the three grids and is spatially related to north-south trending fault zones (Figure 9.2). In these areas, copper is also correlated with anomalous As, U and La. Detailed infill sampling was recommended over these areas, as shown in Figure 9.2.

**Figure 9.2 Regional Copper Soil Geochemistry in the Kachindu Licence Showing Areas Requiring Infill Sampling, Overlain on the Published 1971 Geological Map of the Area**

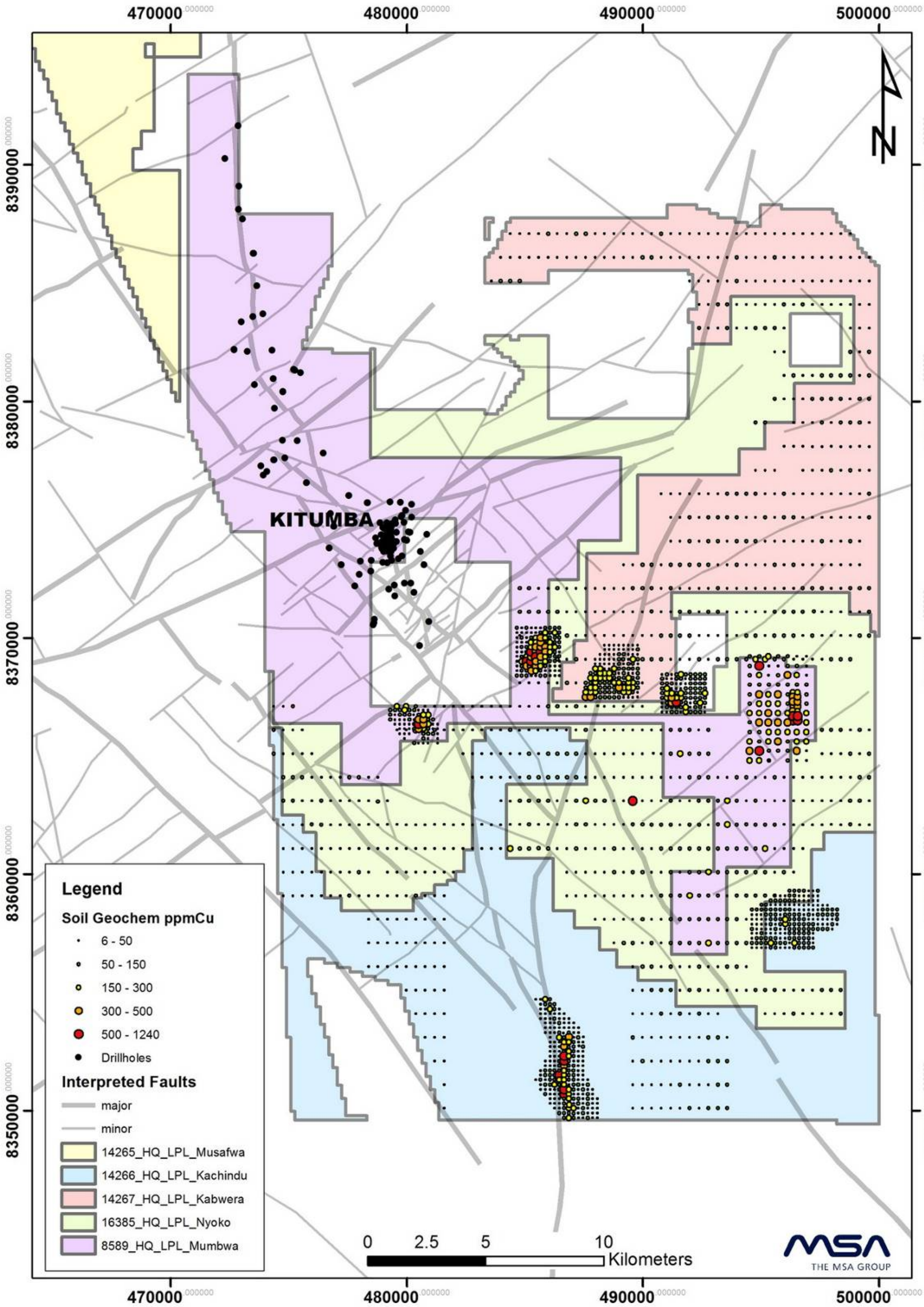


## 9.5 Soil Geochemical Sampling - 2014

An extensive regional soil geochemical sampling programme is being carried out as part of the 2014 Phase 8 work programme, to cover previously untested areas within the wider Mumbwa tenement package. The soil geochemical survey is being carried out by GeoQuest from Lusaka, Zambia. Initial regional sampling has been completed along 1 km spaced east-west lines with a sample spacing of 400 m (Figure 9.3). Subsequent infill sampling is being carried out in anomalous areas. Sampling has confirmed the north-south trending copper soil anomaly in the south central part of the Kachindu licence defined in the 2012 soil geochemical programme. As at the effective date of this report a total of 2,400 samples had been collected with results shown in (Figure 9.3).

Samples are collected from the top of the B horizon at a nominal depth of 30 cm to 40 cm, placed in plastic sample bags, and subsequently screened to -2 mm and analysed using an InnovX portable XRF spectrometer at the Kitumba camp. Quality control samples are inserted at a frequency of 1 in 10 routine samples and comprise either a duplicate, blank or standard. Observations are recorded at each sample site onto standardised field sheets and include sample number, GPS location, soil horizon, depth, depositional / erosional / dambo regime, landscape setting, soil colour, soil type, soil moisture, slop direction, vegetation, presence of termitaria, and general observations.

**Figure 9.3 Regional and Infill Soil Geochemistry**



## 9.6 Ground Geophysics

### 9.6.1 Orion 3D Survey

An Orion 3D DC-IP-MT (direct current resistivity – induced polarisation – magnetotelluric) survey was carried over the greater Kitumba area in July 2012. The objective of the survey was to detect chargeable zones associated with mineralisation and to measure resistivity in order to map structure, alteration and lithology. The survey area was extended beyond the Kitumba deposit in order to define new target areas.

The Orion 3D system acquires three types of geophysical data – magnetotelluric (MT), direct current resistivity (DC) and induced polarisation (IP). The MT and DC methods are used to resolve the resistivity distribution of the subsurface. Resistivity can be an indicator of metallic mineralisation but is more often than not controlled by rock porosity and is therefore an indirect indicator of alteration and lithology. The IP method measures the subsurface electrical capacitance or chargeability, which is a near-direct indicator of the presence of sulphide mineralisation in both disseminated and massive forms. Chargeable zones can however also reflect the presence of graphite and clay minerals.

The DC-IP component of the survey is capable of delineating zones of chargeability within the top 500 m to 750 m from surface, whereas the MT resistivity is useful for mapping geological contacts with resistivity contrasts and conductors (potential alteration or mineralisation) to depths exceeding 1 km.

The survey was conducted by Quantec Geoscience Ltd and covered a rectangular survey grid of 3 km x 2.1 km in extent (Figure 9.4). The survey utilised a pole-dipole configuration with 100 m receiver dipoles and current injection points designed to provide complete sampling of the survey area. Parallel rows and columns of receivers were spaced 300 m apart.

The DC and IP data were inverted to produce 3D models of the subsurface resistivity and chargeability. Final inversion models of the data were presented graphically in Geosoft voxel format along with an interpretation overlay and comments on the most significant results and recommended targets. The final report compiled by Killin (2012) also contained east-west and north-south sections at 25 m intervals across the entire survey area, as well as elevation plan maps at 25 m depth intervals.

Lineaments in the DC resistivity data were identified from elevation plan maps and combined to produce a 3D structural model (Figure 9.5). The Kitumba deposit correlates with an obvious chargeability anomaly as shown in section in Figure 9.6. Eight areas of interest, based mainly on the chargeability model, were selected for potential follow-up and are shown in the plan map in Figure 9.7. One of these (Area C) represents the Kitumba deposit. Inversion modelling of the MT data did not provide much value in further discriminating conductors.

**Figure 9.4 Orion 3D and Kakozhi IP-Resistivity Survey Lines Overlain on the FALCON® Vertical Gravity Gradient (GDD) Imagery**

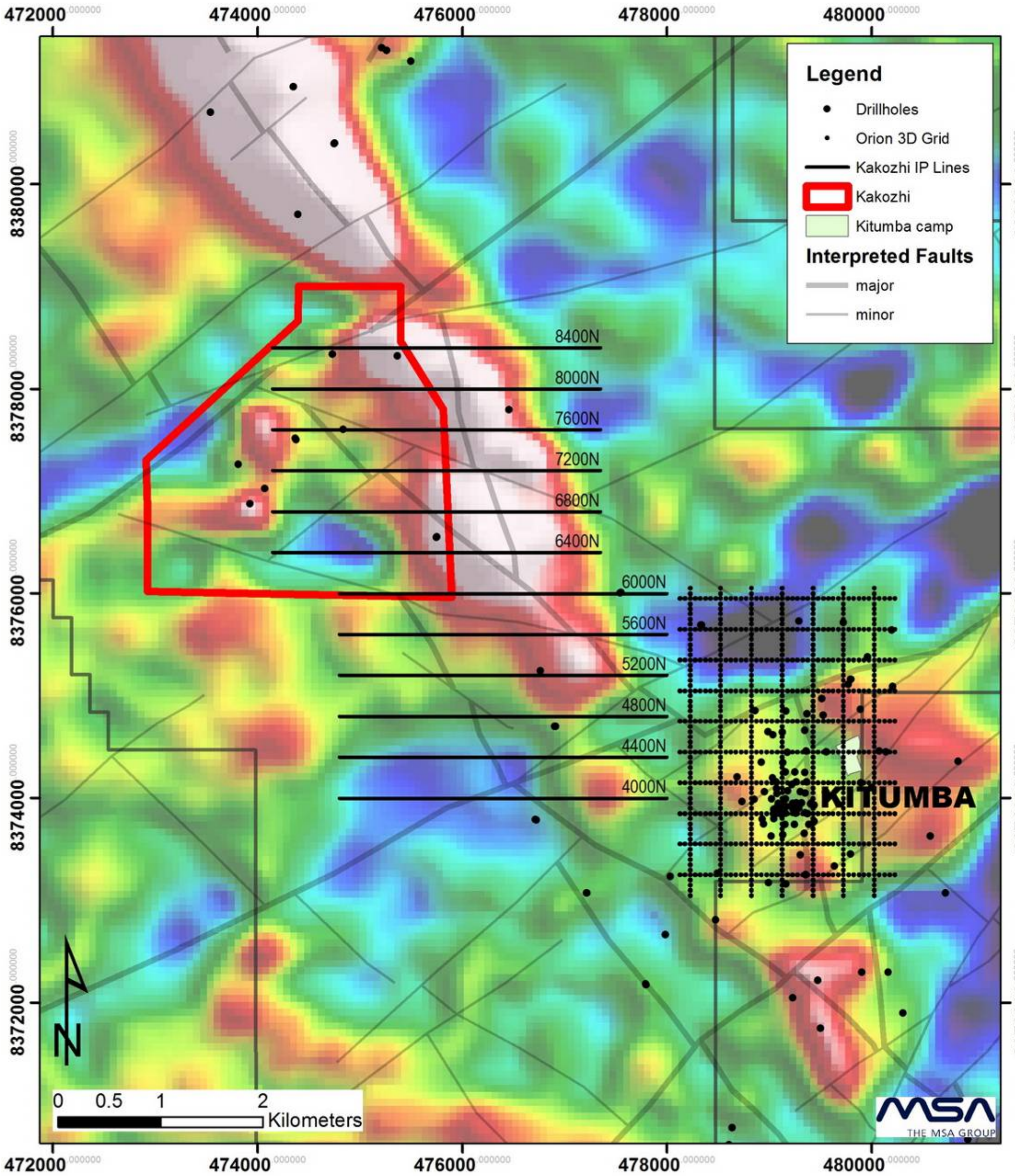


**Figure 9.5 Interpreted 3D Lineaments and Resistivity Model (at 950 m elevation). Source: Killin (2012)**

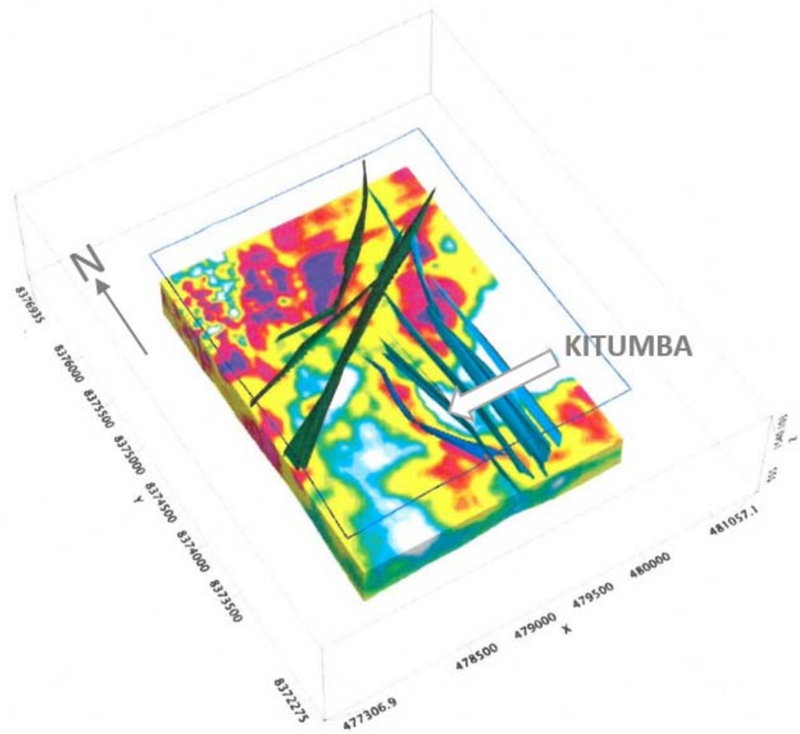



**Figure 9.6 North-south (left) and east-west (right) Chargeability Sections through the Kitumba Deposit (designated as C). Source: Killin (2012)**

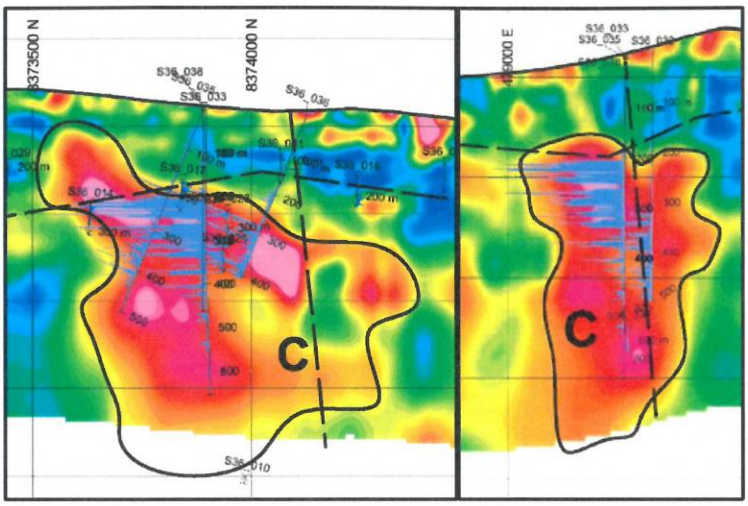
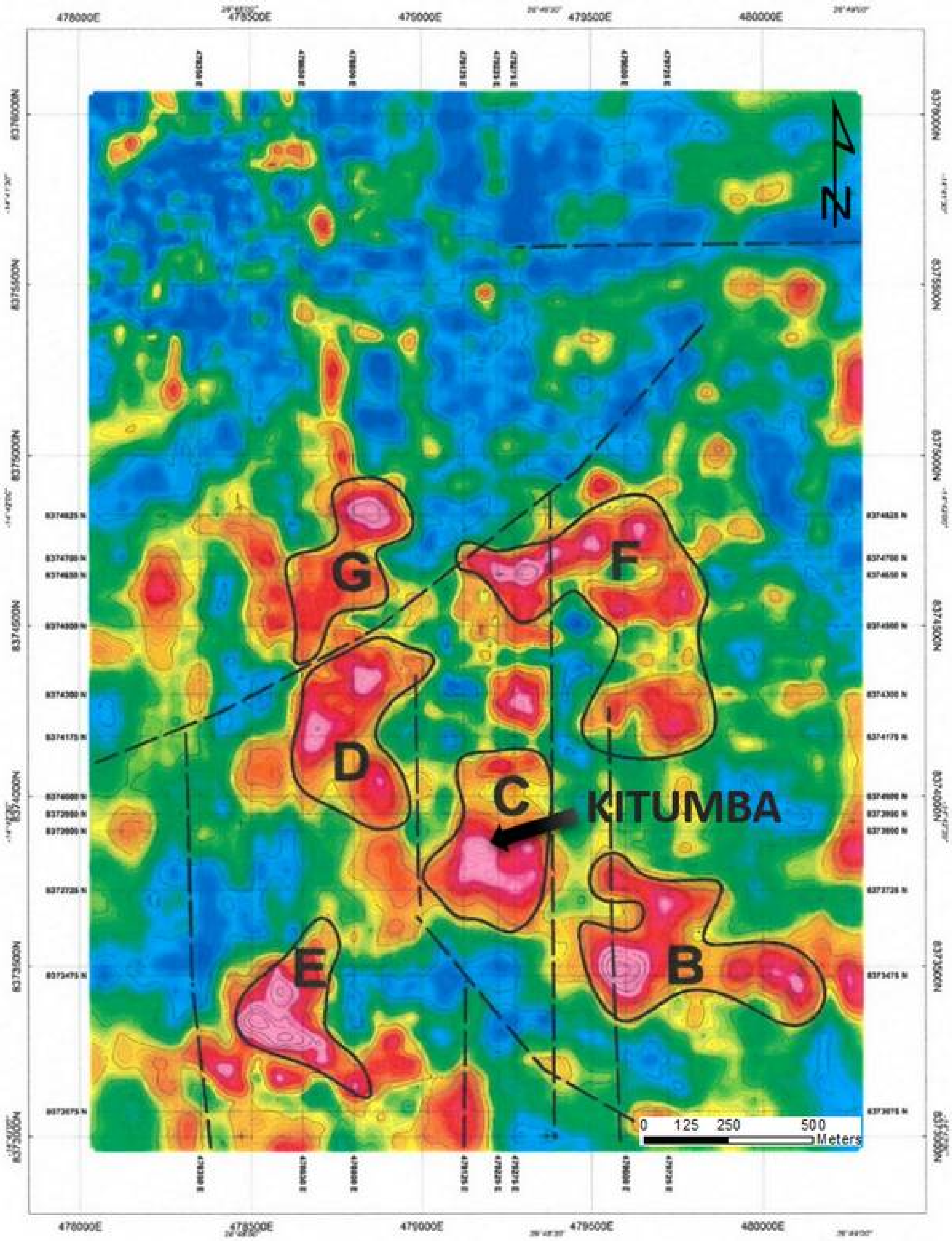


**Figure 9.7 IP Chargeability Plan Map at 1,000 m Elevation Showing Chargeability Anomalies. Source: Killin (2012).**



### 9.6.2 Kakozhi IP-R Survey

An Induced Polarisation – Resistivity (IP-R) survey was completed in August 2014 over the Kakozhi area and including the area between Kakozhi and Kitumba. The objective of the survey was to map zones of chargeability and conductivity with a view to defining potential mineralisation and to obtain better understanding of the structure. The survey was carried by Spectral Geophysics and comprised 12 east-west lines 400 m apart (Figure 9.4).

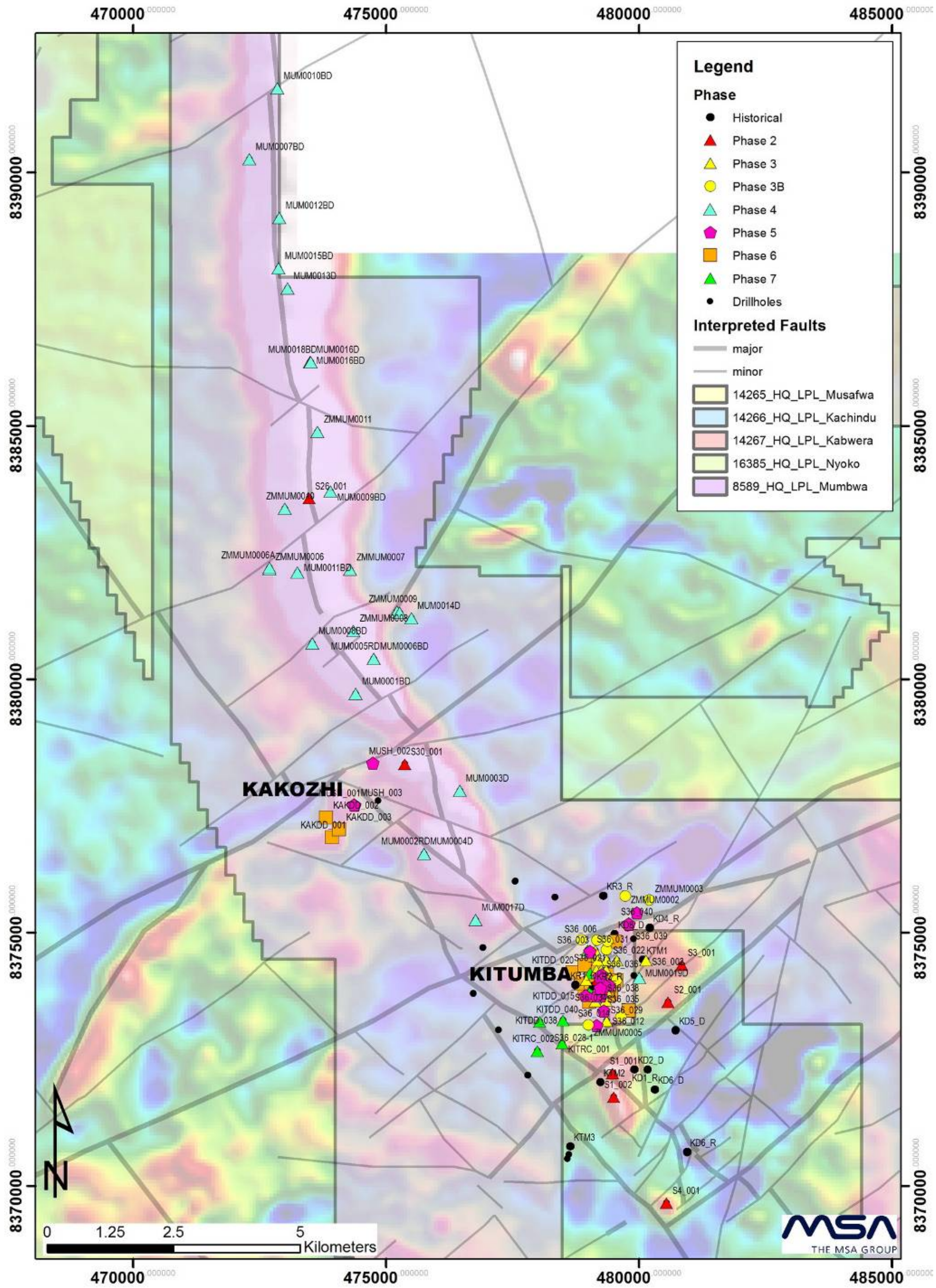
The survey effectively extends the understanding gained from the earlier Orion 3D survey, and covers the jog or offset continuation of the Mumbwa Fault Zone to the north-northwest.

As at the date of this report, the outcomes of this survey had not yet been finalised and therefore are not reported herein.

## 10.0 DRILLING

A total of seven drilling campaigns have been conducted on the Mumbwa tenements since 2006, excluding historical holes drilled up to 2001. The location of these holes is shown by drilling phase in Figure 10.1.

**Figure 10.1 Location of Drill Holes by Phase overlain on Gravity Gradient Imagery**



## 10.1 Phase 1 (2004 – 2006)

In 2004, AIM Resources Limited (later Blackthorn) commissioned a FALCON® gravity gradiometer survey including acquisition of magnetic and radiometric data. In 2006, the gravity potential field data were modelled with 3D inversions after which both data and inversion models were subject to a neural-network analysis. Based on these results, 11 drill holes constituting 5,700 m of drilling were proposed for drilling during 2006 / 2007.

## 10.2 Phase 2 (2006 – 2007)

Drilling commenced in June 2006 and of the planned drill targets eight holes constituting 4,105 m were drilled as shown in Table 10.1. Of the remaining three holes planned, two were abandoned after the first hole on that anomaly yielded disappointing results whilst poor drilling progress precluded drilling of the final drill hole before heavy seasonal rains started.

Laboratory results for the drilling programme were reported during the wet season. Significant mineralisation was intersected in drill hole S36_001 (considered the 'discovery hole' at Kitumba) with 655 m at 0.46% Cu from 42 m depth (at a cut-off of 0.25% Cu).

**Table 10.1 Phase 2 Drilling Summary**

| Hole ID | Easting | Northing | Azimuth | Dip | Elevation | EOH |
|---|---|---|---|---|---|---|
| S1-001 | 479 452.20 | 8 372 200.80 | 0 | -90 | 1 346.70 | 499.15 |
| S1-002 | 479 497.80 | 8 371 741.10 | 0 | -90 | 1 346.60 | 500.6 |
| S3-001 | 480 813.40 | 8 374 339.70 | 0 | -90 | 1 380.60 | 520.8 |
| S26-001 | 473 452.00 | 8 383 548.20 | 0 | -90 | 1 164.10 | 390.75 |
| S30-001 | 475 341.20 | 8 378 302.50 | 0 | -90 | 1 235.70 | 500.27 |
| S4-001 | 480 515.80 | 8 369 647.20 | 0 | -90 | 1 374.20 | 500.5 |
| S36-001 | 479 152.80 | 8 374 052.70 | 90 | -70 | 1 413.50 | 697.4 |
| S2-001 | 480 570.70 | 8 373 624.50 | 0 | -90 | 1 389.60 | 495.7 |
| | | | | | **Total** | **4 105.17** |

## 10.3 Phase 3 (2007 – 2008)

A follow-up drilling programme was designed in September 2007 to systematically expand on the results intersected by drillhole S36_001. Initially 15 holes were proposed for a total of 10,500 m, with hole spacing on a 200 m east-west grid and a 400 m north-south grid, with an average hole depth of 700 m.

During this programme, the proposed hole locations were revised in the light of ongoing drilling progress and a review of the geochemical results from the Phase 2 programme. The drillhole spacing was reduced to 100 m in an east-west direction and 200 m in a north-south direction. In addition, it was decided to reduce the proposed average depth of the holes to 400 m.

A total of 8,000 m were completed in Phase 3 (Table 10.2). Summary results from Phase 3 drilling are reported in Section 10.11.

**Table 10.2 Phase 3 Drilling Summary**

| Hole ID | Easting | Northing | Azimuth | Dip | Elevation | EOH |
|---|---|---|---|---|---|---|
| S36_007 | 479 352.65 | 8 374 450.40 | 90 | -70 | 1 422.42 | 662 |
| S36_009 | 479 152.63 | 8 374 450.45 | 90 | -70 | 1 407.79 | 792 |
| S36_010 | 478 951.77 | 8 374 050.45 | 90 | -70 | 1 402.99 | 866.65 |
| S36_014 | 479 150.95 | 8 373 650.46 | 90 | -70 | 1 443.27 | 594 |
| S36_008 | 479 351.81 | 8 374 050.46 | 90 | -70 | 1 454.33 | 196.5 |
| S36_002 | 480 137.24 | 8 374 453.00 | 90 | -70 | 1 357.29 | 407.7 |
| S36_011 | 479 350.97 | 8 373 650.41 | 90 | -70 | 1 492.63 | 515 |
| S36_016 | 479 152.79 | 8 374 253.00 | 90 | -70 | 1 429.85 | 438.5 |
| S36_020 | 479 252.79 | 8 374 253.00 | 90 | -70 | 1 441.16 | 220 |
| S36_021 | 479 352.79 | 8 374 253.00 | 90 | -70 | 1 451.65 | 400 |
| S36_017 | 479 152.79 | 8 373 852.72 | 90 | -70 | 1 429.43 | 500.5 |
| S36_015 | 479 252.79 | 8 374 052.72 | 90 | -70 | 1 445.47 | 355.5 |
| S36_018 | 479 253.00 | 8 373 853.00 | 90 | -70 | 1 451.07 | 332 |
| S36_012 | 479 350.12 | 8 373 250.36 | 90 | -70 | 1 490.41 | 450.5 |
| S36_003 | 479 553.00 | 8 374 453.00 | 90 | -70 | 1 410.63 | 432 |
| S36_004 | 479 153.00 | 8 374 653.00 | 90 | -70 | 1 399.13 | 400 |
| *S36_005* | *479 353.00* | *8 374 653.00* | *90* | *-70* | *1 426.44* | *270* |
| *S36_006* | *479 153.00* | *8 374 853.00* | *90* | *-70* | *1 416.77* | *167.15* |
| | | | | | **Total** | **8 000.00** |

*Note: S36_005 and S36_006 were completed as part of the Phase 3B programme*

## 10.4 Phase 3B (2008 – 2009)

BHP Billiton assumed operational control of the project on 19 August 2008 and continued with the Phase 3B programme comprising 2,895 m remaining from the previous phase. In total 6,712 m were drilled during Phase 3B from 11 holes (Table 10.3).

**Table 10.3 Phase 3B Drilling Summary**

| Hole ID | Easting | Northing | Azimuth | Dip | Elevation | EOH |
|---|---|---|---|---|---|---|
| S36_005 | 479 353 | 8 374 653 | 90 | -70 | | 490 |
| S36_006 | 479 153 | 8 374 853 | 90 | -70 | | 665.6 |
| S36_013 | 479 553 | 8 374 053 | 270 | -70 | | 453 |
| S36_022 | 479 370 | 8 374 823 | 90 | -70 | | 851.5 |
| S36_013A | 479 553 | 8 374 058 | 90 | -70 | | 412.5 |
| MUS-001 | 472 693 | 8 382 158 | 90 | -60 | | 167.19 |
| ZMMUM0001 | 478 863 | 8 374 853 | 90 | -60.1 | 1391 | 1004.5 |
| ZMMUM0002 | 479 723 | 8 375 712 | 268.18 | -61 | 1321 | 572.8 |
| ZMMUM0003 | 480 197 | 8 375 638 | 276.08 | -60.9 | 1343 | 430.4 |
| ZMMUM0004 | 479 638 | 8 373 335 | 360 | -60 | 1444 | 932.65 |
| ZMMUM0005 | 478 992 | 8 373 169 | 360 | -60 | 1415 | 732 |
| | | | | | **Total** | **6 712.14** |

## 10.5 Phase 4 (2009-2010)

Phase 4 commenced in July 2009 and continued until December 2010 with a hiatus between September 2009 and May 2010 attributed to the annual wet season and sourcing drilling equipment. This drilling phase targeted the Mushingashi density anomaly which extends 12 km to the northwest from the Mutoya target area to the Mushingashi target area. The objective of the Phase 4 programme was to test the anomaly for a potential copper resource. One scout hole was also drilled near the Kitumba Prospect to test a gravity high on the eastern side of the Kitumba hill.

A total of 18,670 m from 26 holes were drilled during Phase 4 (Table 10.4). A summary of assay results is given in Table 10.5. Intersections are generally low grade and/or narrow, with some at significant depths. These intersections also lie below at least 300 m of Karoo Supergroup cover which overlies the Mushingashi area.

**Table 10.4 Phase 4 Drilling Summary**

| Hole ID | Easting | Northing | Azimuth | Dip | Elevation | End of Hole |
|---|---|---|---|---|---|---|
| ZMMUM0006 | 472 691 | 8 382 160 | 90 | -60 | 1168 | 119.5 |
| ZMMUM0006A | 472 693 | 8 382 188 | 90 | -60 | 1168 | 718.2 |
| ZMMUM0007 | 474 291 | 8 382 154 | 270 | -60 | 1185 | 750.4 |
| ZMMUM0008 | 474 352 | 8 380 949 | 0 | -90 | 1168 | 668.3 |
| ZMMUM0009 | 475 215 | 8 381 328 | 122 | -50 | 1203 | 26.5 |
| ZMMUM0009A | 475 265 | 8 381 304 | 122 | -60 | 1203 | 125.85 |
| ZMMUM0010 | 472 994 | 8 383 356 | 135 | -60 | 1163 | 137 |
| ZMMUM0011 | 473 645 | 8 384 872 | 270 | -60 | 1159 | 125 |
| MUM0001BD | 474 398 | 8 379 700 | 90 | -60 | 1241 | 850.7 |
| MUM0002RD | 475 753 | 8 376 550 | 112 | -60 | 1205 | 63 |
| MUM0003D | 476 459 | 8 377 794 | 270 | -50 | 1247 | 1032 |
| MUM0004D | 475 748 | 8 376 545 | 112 | -60 | 1205 | 1077.8 |
| MUM0005RD | 474 758 | 8 380 398 | 270 | -60 | 1116 | 174 |
| MUM0006BD | 474 753 | 8 380 393 | 270 | -60 | 1116 | 1004 |
| MUM0007BD | 472 301 | 8 390 250 | 90 | -65 | 1286 | 1026.3 |
| MUM0008BD | 473 545 | 8 380 699 | 90 | -70 | 1112 | 1094.6 |
| MUM0009BD | 473 903 | 8 383 697 | 270 | -60 | 1172 | 1000 |
| MUM0010BD | 472 850 | 8 391 650 | 270 | -90 | 1142 | 911.6 |
| MUM0011BD | 473 249 | 8 382 101 | 90 | -70 | 1173 | 928.3 |
| MUM0012BD | 472 888 | 8 389 098 | 320 | -85 | 1157 | 986.2 |
| MUM0014D | 475 499 | 8 381 199 | 270 | -50 | 1210 | 939.5 |
| MUM0015BD | 472 870 | 8 388 100 | 200 | -80 | 1148 | 1038.7 |
| MUM0016BD | 473 498 | 8 386 253 | 270 | -75 | 1198 | 431.6 |
| MUM0017D | 476 765 | 8 375 240 | 90 | -60 | 1246 | 1100.8 |
| MUM0018BD | 473 510 | 8 386 250 | 270 | -75 | 1164 | 1122 |
| MUM0019D | 480 000 | 8 374 099 | 120 | -65 | 1376 | 1219 |
| | | | | | **Total** | **18 670.85** |

**Table 10.5 Phase 4 Intersection Summary (Cut-off 0.25% Cu; Thickness >2 m)**

| Hole ID | From (m) | To (m) | Drilled Width (m) | %Cu |
|---|---|---|---|---|
| ZMMUM-005 | 246.00 | 258.00 | 12.00 | 0.51 |
| ZMMUM-006A | 490.00 | 493.00 | 3.00 | 0.35 |
| ZMMUM-008 | 300.00 | 303.00 | 3.00 | 0.45 |
| | 306.00 | 310.00 | 4.00 | 0.52 |
| | 340.00 | 346.00 | 6.00 | 0.83 |
| MUM0001BD | 506.00 | 510.00 | 4.00 | 0.26 |
| | 532.00 | 536.00 | 4.00 | 0.28 |
| | 542.00 | 546.00 | 4.00 | 0.44 |
| MUM0003D | 1020.00 | 1026.00 | 6.00 | 0.45 |
| MUM0004D | 510.00 | 514.00 | 4.00 | 0.79 |
| MUM0006BD | 922.00 | 926.00 | 6.00 | 0.28 |
| MUM0007BD | 534.00 | 538.00 | 4.00 | 1.13 |
| | 598.00 | 604.00 | 6.00 | 1.16 |
| | 758.00 | 768.00 | 10.00 | 1.80 |
| | 772.00 | 782.00 | 10.00 | 0.93 |
| | 790.00 | 794.00 | 4.00 | 0.95 |
| | 800.00 | 806.00 | 6.00 | 0.53 |
| MUM0008BD | 347.00 | 351.00 | 4.00 | 0.30 |
| MUM0010BD | 328.00 | 330.00 | 2.00 | 0.33 |
| MUM0011BD | 632.00 | 644.00 | 12.00 | 0.40 |
| MUM0012BD | 604.00 | 614.00 | 10.00 | 0.31 |
| MUM0014D | 372.00 | 378.00 | 6.00 | 1.22 |
| | 388.00 | 398.00 | 10.00 | 0.43 |
| MUM0015D | 430.00 | 436.00 | 6.00 | 0.53 |
| | 468.00 | 472.00 | 4.00 | 0.29 |
| | 518.00 | 522.00 | 4.00 | 0.46 |
| | 548.00 | 572.00 | 24.00 | 0.25 |
| MUM0019D | 516.00 | 518.00 | 2.00 | 0.52 |

## 10.6 Phase 5 (2011-2012)

The major objectives of the Phase 5 drilling programme were to:

- Expand the maiden Mineral Resource estimate stated in December 2009, through a programme of ‘step-out’ drilling.
- Increase the confidence level of the Mineral Resource estimate and upgrade parts of the Inferred Mineral Resource to at least Indicated status, through a programme of infill drilling.
- Test the prioritised exploration targets at Target A (Kakozhi).

The Phase 5 drilling programme commenced in August 2011 and completed drilling of 21 diamond core holes for a total of 10 934 m as at April 2012 (Table 10.6).

Initial drilling focussed on infilling the existing 200 m x 200 m spaced Phase 3 holes to a nominal 100 m spacing. On confirming a high-grade zone in the southern part of the Kitumba deposit (intersected in part by S36_001 in Phase 2 and S36_017 and S36_018 in Phase 3), subsequent infill drilling was carried out on a 50 m grid.

Diamond core drilling commenced on 21 August 2011 and 2 301 m was completed by 12 December 2012. Holes were drilled HQ size (63.5 mm diameter) to hard rock and thereafter NQ size (47.6 mm diameter). The initial drilling contractor, Drillcorp Africa Pty Ltd was replaced due to poor performance. Ox Drilling Limited (Ox) was contracted to complete the Phase 5 programme and commenced drilling on 21 November 2012 and completed 8,633 m by 22 April 2012. Holes were drilled PQ size (108 mm diameter) through overburden. Core size was reduced to HQ in hard rock and thereafter NQ once most of the oxidised zone had been traversed. The majority of core was drilled NQ size.

Significant Phase 5 intersections are reported in Section 10.9 (Target A/Kakozhi 'scout' holes) and Section 10.11.

**Table 10.6 Phase 5 Drilling Summary**

| Type | | Target ID | Drillhole ID | Planned EOH Depth (m) | Actual EOH Depth (m) | UTM Zone 35S WGS84 | | | Planned Inclination | Planned Azimuth |
|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | Planned Easting (m) | Planned Northing (m) | Planned RL (m) | | |
| Infill | 1 | P5_001 | S36-023 | 500.00 | 483.05 | 479,310 | 8,373,955 | 1,440 | -70 | 270 |
| | 2 | P5_002 | S36-024 | 500.00 | 583.48 | 479,240 | 8,373,750 | 1,430 | -90 | 000 |
| | 3 | P5_003 | S36-026* | 500.00 | 614.82 | 479,260 | 8,374,160 | 1,430 | -90 | 000 |
| | 4 | P5_005 | S36-027 | 500.00 | 509.00 | 479,140 | 8,373,750 | 1,430 | -90 | 000 |
| | 5 | P5_007 | S36_029 | 600.00 | 600.80 | 479,300 | 8,373,440 | 1,500 | -70 | 270 |
| | 6 | P5_006 | S36_028 | 500.00 | 524.46 | 479,160 | 8,374,160 | 1,430 | -90 | 000 |
| | 7 | P5_009 | S36-025 | 500.00 | 532.32 | 479,410 | 8,373,955 | 1,463 | -65 | 270 |
| | 8 | P5_101 | S36_032 | 500.00 | 500.50 | 479,280 | 8,373,900 | 1,452 | -90 | 000 |
| | 9 | P5_102 | S36_033** | 500.00 | 463.36 | 479,230 | 8,373,900 | 1,442 | -90 | 000 |
| | 17 | | S36_038 *** | 600.00 | 653.55 | | | | | |
| | 10 | P5_103 | S36_034 | 500.00 | 500.55 | 479,310 | 8,373,955 | 1,453 | -90 | 000 |
| | 16 | P5_104 | S36_035 | 500.00 | 500.20 | 479,230 | 8,373,900 | 1,435 | -70 | 180 |
| | 18 | P5_105 | S36_036 | 600.00 | 653.54 | 479,219 | 8,374,141 | 1,442 | -70 | 180 |
| Step-out | 19 | P5_014 | S36_037 | 600.00 | 653.43 | 479,947 | 8,375,383 | 1,335 | -60 | 325 |
| | 20 | P5_013 | S36_039**** | 600.00 | 186.85 | 479,791 | 8,375,159 | 1,405 | -70 | 325 |
| | 21 | | S36_040***** | 600.00 | 602.55 | | | | | |
| | 11 | P5_010 | S36_030 | 500.00 | 506.50 | 478,940 | 8,373,750 | 1,430 | -80 | 090 |
| | 12 | KIT_03 | S36_031 | 500.00 | 500.20 | 479,035 | 8,374,622 | 1,400 | -60 | 325 |
| Scout | 13 | Target A-01 | MUSH_001 | 800.00 | 202.35 | 474,394 | 8,377,510 | 1,318 | -70 | 270 |
| | 14 | | MUSH_003 | | 563.00 | | | | | |
| | 15 | Target A-02 | MUSH_002 | 500.00 | 600.00 | 474,731 | 8,378,340 | 1,294 | -70 | 045 |
| | | | **TOTAL** | **10900.00** | **10934.51** | | | | | |

* *Drillhole S36_026 was completed to an EOH depth = 516.10m in 2011*
*In January 2012 this hole was extended to EOH = 614.82m following observation of mineralisation at 516m interval.*
** *Drill hole S36_033 was abandoned at 463.36m due to drilling rods becoming trapped and equipment lost in the hole.*
*** *Replacement drill hole S36_038 is redrilling target from surface as a new hole to a deeper depth.*
**** *Drill hole S36_039 was abandoned at 186.85m due to problems in hole with drilling equipment.*

## 10.7 Phase 6 (2012-2013)

Drilling commenced on 22 June 2012 with two RC holes in search of a new water supply borehole. These boreholes were sighted on untested targets identified by a Knight Piésold study. Only the first hole, KITRC_001, intersected significant water at 96 m where carbonaceous sediments were intersected, this is now the water supply borehole for drilling. The second borehole failed to intersect significant water.

Diamond drilling commenced on 2 July 2012 and was completed on 26 February 2013 for a total of 16,223 m. Drillhole coverage after the Phase 6 drilling is show in Figure 10.2. A summary of the Phase 6 drilling appears in Table 10.7.

Initial drilling focussed on testing the potential for a shallow resource at Kitumba. Later drilling was planned to extend and upgrade the existing Mineral Resource through infill and step-out drilling and testing of targets defined by the Orion 3D survey undertaken between 28 June and 23 July 2012.

Diamond core drilling was undertaken by Ox using three Boart Longyear LF90D rigs and one Boart Longyear LF230 rig. Core size through the overburden was PQ which was reduced to HQ in hard rock and thereafter NQ once the oxidised zone had been traversed. Exceptions were drill holes identified for metallurgical testwork, which were drilled at PQ and HQ sizes to provide sufficient material for the testwork. The majority of core in these holes was drilled HQ size.

All holes were surveyed using a Reflex instrument except for those intervals where casing was lost in the hole. These holes were individually assessed and it was determined that the missing survey intervals would not compromise the reliability of the Mineral Resource estimate. Routine survey shots were taken at 6 m intervals for inclined holes and 12 m intervals for vertical holes, which allowed for spurious readings to be discarded based on magnetic field strength and magnetic dip reported by the survey tool. QAQC procedures included testing the survey tool in an orientation test jig.

Accurate collar surveys were carried out by a registered land surveyor using a differential GPS. All surveys were reported in UTM Zone 35S WGS84.

Following remodelling of FALCON® gravity and magnetic data in Phase 5 and soil sampling in the Kakozhi area, exploration targets to the northwest of Kitumba were defined. Drilling carried out at Kakozhi in Phase 6 was as a result of this work and in follow-up to the Phase 5 'scout' drilling and is discussed in Section 10.9.

Significant Phase 6 intersections are reported in Section 10.9 (Target A/Kakozhi) and Section 10.11.

**Figure 10.2 Drillhole Coverage of the Mineral Resource Area after Completion of the Phase 6 Drilling**

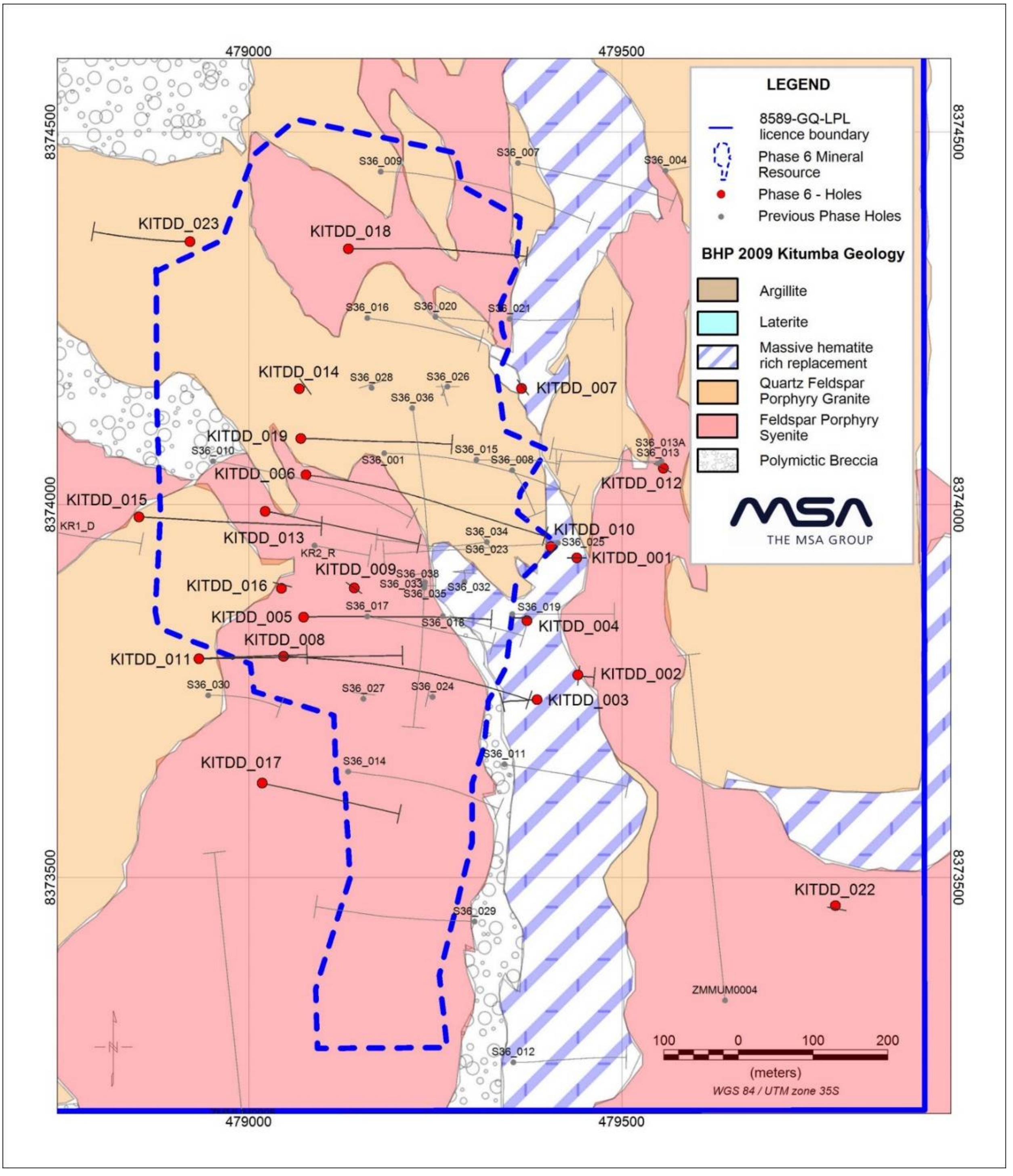


**Table 10.7 Phase 6 Drilling Summary**

| Type | Drillhole ID | Planed EOH Depth (m) | Actual EOH Depth (m) | UTM Zone 35S WGS84 | | | Inclination | Azimuth |
|---|---|---|---|---|---|---|---|---|
| | | | | Easting | Northing | RL | | |
| Exploration | KAKDD_001 | 600 | 614 | 473927.353 | 8376871.975 | 1406.253 | -70 | 270 |
| | KAKDD_002 | 600 | 671.5 | 473815.518 | 8377258.53 | 1302.994 | -70 | 90 |
| | KAKDD_003 | 600 | 664.9 | 474072.521 | 8377024.702 | 1420.601 | -70 | 45 |
| In-fill | KITDD_003 | 600 | 602.65 | 479386.097 | 8373738.382 | 1493.218 | -85 | 270 |
| | KITDD_004 | 600 | 598.5 | 479372.617 | 8373844.312 | 1480.044 | -90 | 0 |
| | KITDD_005 | 600 | 644.5 | 479073.238 | 8373849.029 | 1411.712 | -65 | 90 |
| | KITDD_006 | 600 | 725.36 | 479076.455 | 8374040.039 | 1403.27 | -65 | 90 |
| | KITDD_007 | 600 | 548.5 | 479365.375 | 8374155.945 | 1449.502 | -90 | 0 |
| | KITDD_010 | 600 | 620.1 | 479404.728 | 8373943.681 | 1479.005 | -90 | 0 |
| | KITDD_013 | 600 | 645.5 | 479021.673 | 8373990.997 | 1399.578 | -70 | 90 |
| | KITDD_019 | 600 | 632.5 | 479069.488 | 8374088.77 | 1409.073 | -70 | 90 |
| | S36_032 | 700 | 586.2 | 479289.01 | 8373896.17 | 1451 | -90 | 0 |
| Satellite target | KITDD_020 | 450 | 457.44 | 478688.18 | 8374204.17 | 1384 | -90 | 0 |
| | KITDD_021 | 600 | 621.3 | 479529.00 | 8374809.00 | 1439 | -90 | 0 |
| | KITDD_022 | 600 | 476.35 | 479790.00 | 8373454.00 | 1406 | -90 | 0 |
| | KITDD_023 | 600 | 639.24 | 478919.00 | 8374352.00 | 1387 | -80 | 270 |
| Step-out | KITDD_001 | 600 | 332 | 479439.368 | 8373928.613 | 1489.321 | -90 | 0 |
| | KITDD_002 | 600 | 555.85 | 479440.791 | 8373771.487 | 1505.01 | -90 | 0 |
| | KITDD_008 | 800 | 884.14 | 479046.394 | 8373796.494 | 1409.049 | -70 | 90 |
| | KITDD_009 | 700 | 640.1 | 479141.114 | 8373888.196 | 1420.571 | -90 | 0 |
| | KITDD_011 | 800 | 781.52 | 478932.903 | 8373793.228 | 1394.554 | -70 | 90 |
| | KITDD_012 | 400 | 394.6 | 479555.365 | 8374048.872 | 1444.579 | -90 | 0 |
| | KITDD_014 | 600 | 627.3 | 479067.424 | 8374155.579 | 1419.006 | -90 | 0 |
| | KITDD_015 | 800 | 692.65 | 478852.495 | 8373983.557 | 1391.417 | -70 | 90 |
| | KITDD_016 | 600 | 601.2 | 479043.511 | 8373887.734 | 1407.333 | -90 | 0 |
| | KITDD_017 | 600 | 572.3 | 479017.695 | 8373626.367 | 1421.731 | -70 | 100 |
| | KITDD_018 | 600 | 616.2 | 479133.217 | 8374343.376 | 1411.415 | -70 | 90 |
| | **TOTAL** | **16150** | **16223.15** | | | | | |

## 10.8 Phase 7 (2012-2013)

The main objectives of the Phase 7 work programme were to:

- Convert a proportion of the Indicated Mineral Resources in the high grade core of the deposit to Measured Mineral Resources by way of additional infill drilling.

- Assess the potential for further deep hypogene mineralisation as delineated during the Phase 5 and Phase 6 drilling.

- Drill geotechnical holes to further characterise the structural and engineering properties of material within the current extent of potential underground mining operations.

- Test prioritised satellite targets proximal to the Kitumba deposit, defined by Orion 3D geophysical anomalies.

Drilling was undertaken by Ox using a similar approach to the previous phases. The locations of the Phase 7 Kitumba holes are shown in Figure 10.3. A total of 10 infill diamond drill holes were completed for 5 284 m, reducing the overall spacing in the high grade core of the deposit to approximately 30 m. A summary of the Phase 7 drilling appears in Table 10.8. Extension of the deep hypogene mineralisation defined in Phase 5 was tested further through deepening of two Phase 5 holes and drilling of an additional hole from surface for a total of 1,282 m.

All holes were surveyed using a gyro instrument at 5 m down hole intervals. Accurate collar surveys were carried out by a registered land surveyor using a differential GPS. All surveys were reported in UTM Zone 35S WGS84.

Three geotechnical holes were core drilled for a total of 1,664 m and three satellite holes were completed for 1,368 m, the latter to test Orion 3D chargeability targets to the immediate southwest and north of the Kitumba. The chargeability anomalies to the west of the Kitumba deposit were tested by holes KITDD_038 and KITDD_040. Hypogene sulphides were intersected in syenites and metsediments in KITDD_040 with intersections above a 0.25% Cu cut-off of 12 m at 0.55% Cu from 307 m to 319 m and 17 m at 0.42% Cu from 399 m to 416 m. No assay results above cut-off were reported from KITDD_038.

KITDD_039 tested a chargeability high associated with a fault zone to the north-northwest of the Kitumba deposit and returned an intersection of 6 m at 0.29% Cu from 304 m to 310 m.

Significant Phase 7 intersections from the Mineral Resource area are reported in Section 10.11.

**Figure 10.3 Drillhole Coverage of the Mineral Resource Area after Completion of the Phase 7 Drilling**

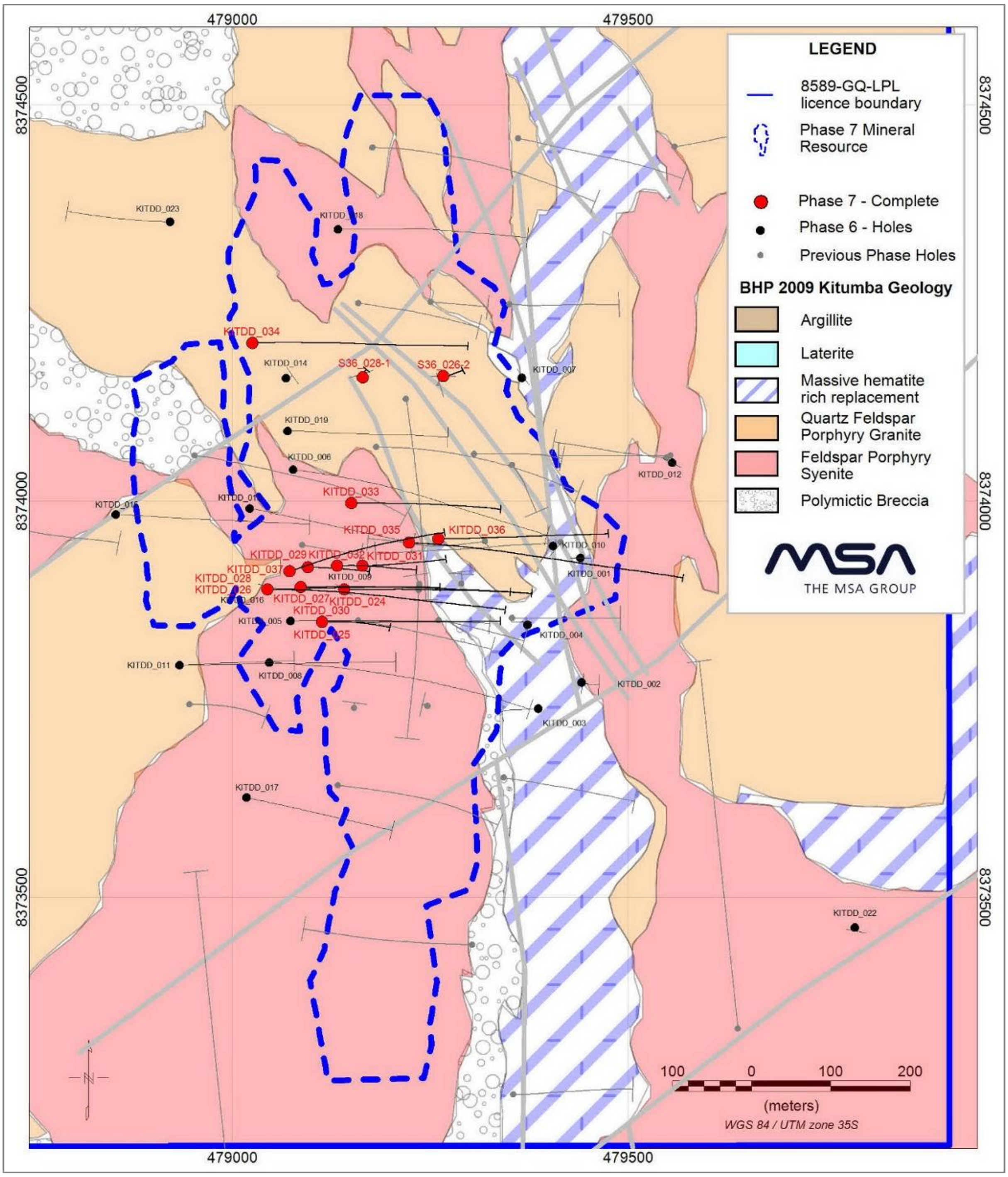


**Table 10.8 Phase 7 Drilling Summary**

| Type | Drillhole ID | EOH | UTM Zone 35S WGS84 | | | Inclination | Azimuth |
|---|---|---|---|---|---|---|---|
| | | | Easting | Northing | RL | | |
| Infill high grade zone | KITDD_024 | 449.70 | 479141 | 8373889 | 1423.8 | -60 | 90 |
| | KITDD_025 | 530.60 | 479113 | 8373848 | 1421.6 | -81 | 90 |
| | KITDD_026 | 557.60 | 479044 | 8373889 | 1410.0 | -68 | 90 |
| | KITDD_027 | 539.90 | 479093 | 8373889 | 1416.4 | -60 | 90 |
| | KITDD_028 | 562.40 | 479044 | 8373889 | 1410.0 | -60 | 90 |
| | KITDD_029 | 419.70 | 479092 | 8373920 | 1413.5 | -80 | 90 |
| | KITDD_030 | 575.80 | 479113 | 8373848 | 1421.6 | -68 | 90 |
| | KITDD_031 | 539.60 | 479173 | 8373920 | 1426.4 | -80 | 90 |
| | KITDD_032 | 581.60 | 479132 | 8373920 | 1420.0 | -80 | 90 |
| | KITDD_033 | 527.50 | 479150 | 8373998 | 1415.6 | -70 | 90 |
| Test deep hypogene mineralisation | S36_026-2 | 614.80-707.20 | 479266 | 8374158 | 1439.0 | -90 | 0 |
| | S36_028-1 | 524.50-986.30 | 479164 | 8374157 | 1428.0 | -90 | 0 |
| | KITDD_034 | 728.50 | 479025 | 8374200 | 1416.0 | -70 | 90 |
| Geotechnical holes | KITDD_035 | 651.00 | 479223 | 8373948 | 1435.0 | -65 | 90 |
| | KITDD_036 | 449.50 | 479260 | 8373948 | 1452.0 | -65 | 90 |
| | KITDD_037 | 563.40 | 479081 | 8373914 | 1413.0 | -70 | 70 |
| Satellite Target | KITDD_038 | 401.60 | 478495 | 8373265 | 1368.0 | -60 | 90 |
| | KITDD_039 | 476.90 | 478987 | 8374646 | 1394.0 | -65 | 270 |
| | KITDD_039-1 | 470.00-543.85 | 478987 | 8374646 | 1394.0 | -65 | 270 |
| | KITDD_040 | 423.20 | 478034 | 8373234 | 1317.0 | -60 | 90 |
| | **TOTAL** | **9663.22** | | | | | |

## 10.9 Phase 8 (Current)

The Phase 8 drilling programme commenced in June 2014 and includes 12,000 m of drilling with the following breakdown:

- 950 m dedicated to metallurgical drilling.
- 2,800 m geotechnical drilling.
- 5,100 m combined metallurgical and resource drilling.
- 1,200 m sterilisation drilling centred on IP anomalies.
- 2,000 m exploration drilling based on the Phase 8 IP survey.

As at the effective date of this report, approximately 8,300 m of drilling had been completed, with assay results pending.

## 10.10 Kakozhi Drilling (2011-2012)

Remodelling of the FALCON® gravity and magnetic data by PGN in 2011 resulted in the definition of several geophysical targets. The priority Kakozhi target is defined by an anomaly with characteristics similar to that at Kitumba, including location at the intersection of faults (along the offset continuation of the Mumbwa Fault Zone), and association with a topographical high. Kakozhi was initially tested during the Phase 5 drill campaign between October and December 2011 by three diamond drill holes for a total of 1 365.35 m. A further three diamond drill holes for a total of 1,950.40 m were completed at Kakozhi during the Phase 6 drilling campaign between July and October 2012, following the Kakozhi geological mapping and soil geochemical sampling programme. The latter three holes were designed to test a coincident gravity anomaly and copper in soil geochemical anomaly and to obtain optimum stratigraphic coverage. Details of the drilling conducted at Kakozhi are shown in Table 10.9. The locations of the holes in relation to the copper soil and gravity anomalies are shown in Figure 10.4.

**Table 10.9 Kakozhi Drilling Summary**

| Phase | Hole ID | Depth | UTM Zone 35S WGS84 | | | Inclination | Azimuth |
|---|---|---|---|---|---|---|---|
| | | | Easting (m) | Northing (m) | RL (m) | | |
| 5 | MUSH_001 | 202.35 | 474380.3 | 8377501.3 | 1294.520 | -70 | 270 |
| | MUSH_002 | 600.00 | 474735.8 | 8378334.5 | 1320.958 | -70 | 45 |
| | MUSH_003* | 563.00 | 474369.2 | 8377513.0 | 1320.755 | -70 | 270 |
| 6 | KAKDD_001 | 614.00 | 473927.4 | 8376872.0 | 1406.253 | -70 | 270 |
| | KAKDD_002 | 671.50 | 473815.5 | 8377258.5 | 1302.994 | -70 | 90 |
| | KAKDD_003 | 664.90 | 474072.5 | 8377024.7 | 1420.601 | -70 | 45 |

*MUSH_001 was abandoned and re-drilled as MUSH_003.

KAKDD_003 was drilled towards an interpreted contact with mineralised syenite and breccia intersected in MUSH_003, which reported the previous best result. Although the hole failed to reach the contact, increasing alteration intensity with depth is interpreted as proximity to the contact.

Selected results for the Kakozhi drilling are reported in Table 10.10. A cut-off grade of 0.25% Cu was applied when delineating the drilled thickness interval of mineralisation, with length-weighted average grades reported. The best copper mineralisation was intersected in KAKDD_001, and although not economically significant, there is evidence for copper enrichment in the area and potential for further exploration. A northeast-southwest section through KAKDD_001 is shown in Figure 10.5.

The results confirm the presence of a mineralised system at Kakozhi, with low grade thick intersections associated with the upper oxidised and ferruginised parts of the holes. These may represent zones of leaching and/or reconcentration of copper mineralisation. Potential exists to discover Kitumba-style supergene-enriched mineralisation and hypogene sulphide mineralisation at Kakozhi. Trace chalcopyrite was logged in KAKDD_002.

**Table 10.10 Selected Kakozhi Drilling Results**

| Hole ID | From (m) | To (m) | Drilled Width (m) | %Cu |
|---|---|---|---|---|
| MUSH_003 | 59.00 | 98.00 | 39.00 | 0.44 |
| | 106.00 | 109.00 | 3.00 | 0.36 |
| | 114.00 | 151.00 | 37.00 | 0.46 |
| | 214.00 | 218.00 | 4.00 | 0.71 |
| KAKDD_001 | 15.90 | 218.44 | 202.54 | 0.35 |
| Including | 147.00 | 178.55 | 31.55 | 0.42 |
| | 185.44 | 194.00 | 8.56 | 0.78 |
| KAKDD_002 | 230.00 | 246.00 | 16.00 | 0.36 |
| | 376.00 | 380.00 | 4.00 | 0.71 |
| KAKDD_003 | 14.00 | 38.00 | 24.00 | 0.37 |
| | 326.00 | 356.00 | 30.00 | 0.36 |
| | 418.00 | 450.00 | 32.00 | 0.76 |

The holes were completed by Ox Drilling Limited using Boart Longyear LF90D rigs. Holes were drilled PQ diameter through the overburden, reduced to HQ through the remainder of the oxidised zone and completed in NQ size. Appropriate measures were undertaken to maximise sample recovery including the use of drilling muds and chemicals.

**Figure 10.4 Plan View Showing Kakozhi Drill Holes with Copper Grade, Outline of the SG = 2.94 Gravity Anomaly and Gridded Copper Soil Geochemistry**

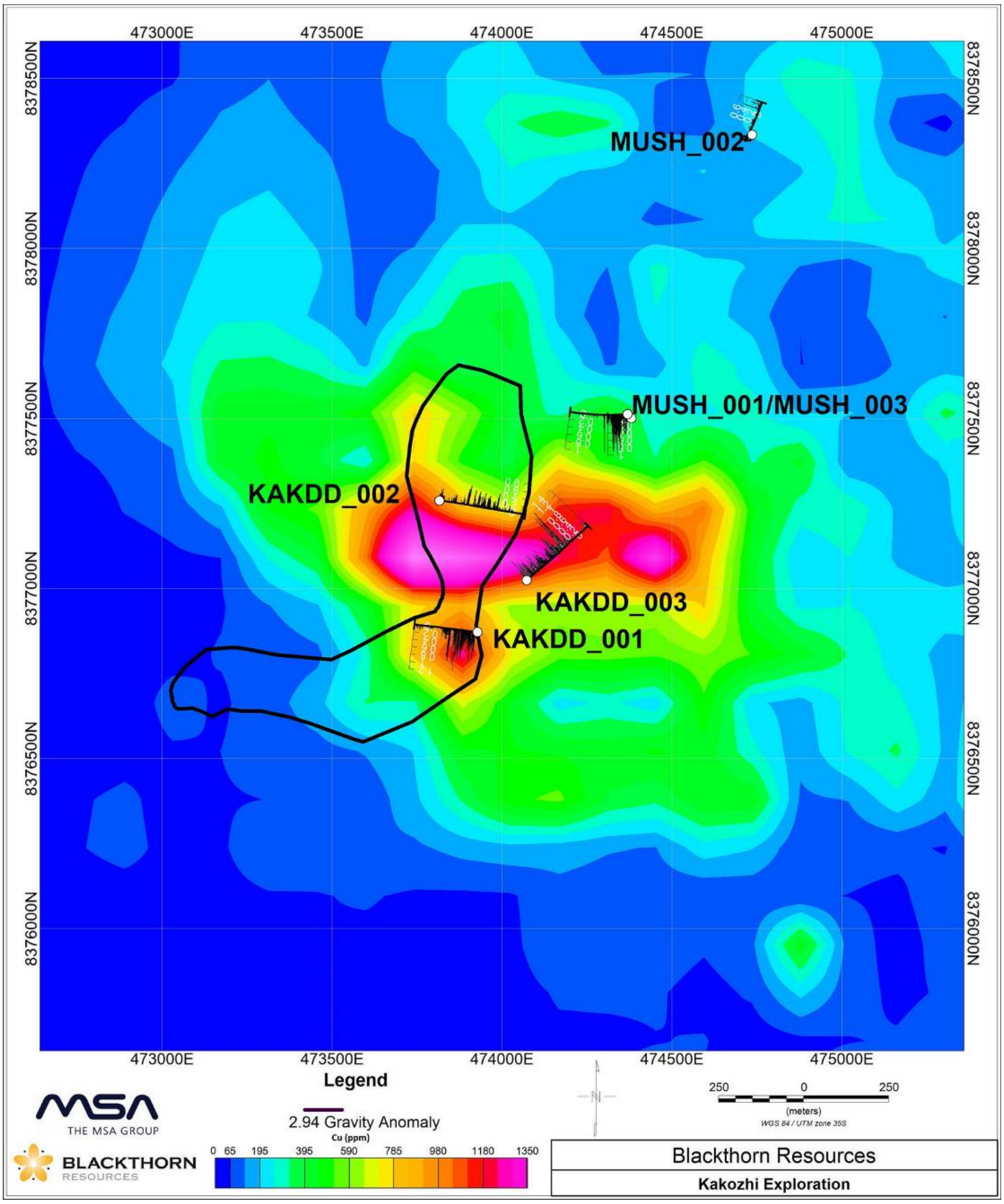
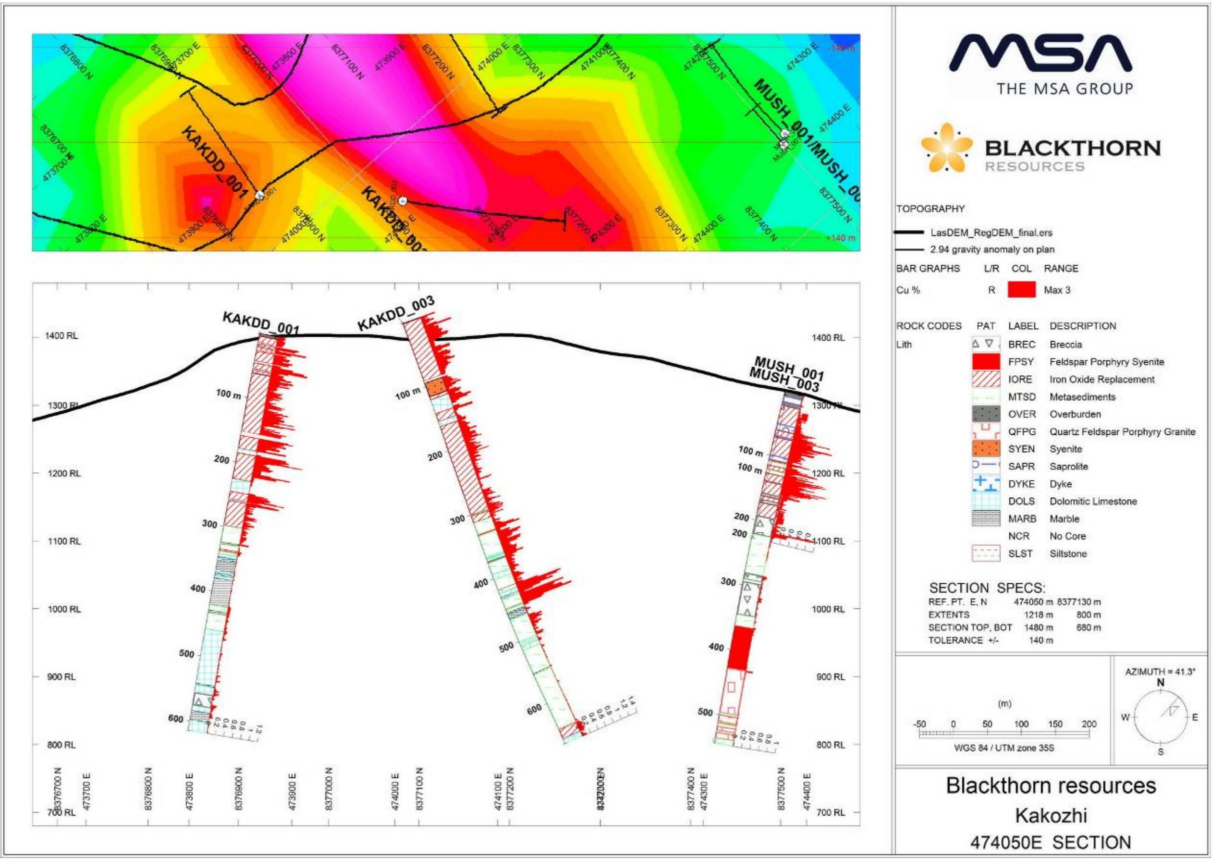


**Figure 10.5 Northeast-southwest Section through Kakozhi Drill Holes**




## 10.11 Logging

### 10.11.1 Lithological Logging

All drilling, logging, sampling, assaying and QAQC was carried out according to the Standard Operating Procedures (SOPs) documented for the Kitumba Copper Project by MSA.

Logging was initially undertaken directly onto pre-designed hardcopy log sheet templates and thereafter captured into MS Excel and emailed to the MSA Data Manager for validation and import into an MS Access project database.

During the course of the project, the database was transferred to a Maxwell Logchief™ ™-Datashed database management system and, from October 2013, all data from the hardcopy log sheets were captured directly into the Logchief™ ™ software which was later synchronised with the Datashed project database in Johannesburg. This had the advantage of on-site validation and more efficient identification and correction of errors. All pre-Phase 6 data was migrated into the Datashed database and was validated.

Logging paid particular attention to characterising lithology, alteration types and textures, mineralisation, mineralisation zone and brecciation, and the inter-relationships between these. Aspects of the logging and sampling at Kitumba are shown in Figure 10.6.

The various drilling phases were staffed by different geologists, resulting in slight differences in logging approach over time. As a result of improved understanding of the Kitumba mineralised system, core from previous holes that were included in the updated MRE was re-logged in order to conform to the updated logging protocols and to ensure standardisation in the database used in the MRE. A total of 30,322 m were re-logged for lithology, alteration, mineralisation and mineralisation zone (including the Phase 6 holes that were completed before the protocols were finalised).

In addition to the routine geological logs, magnetic susceptibly measurements (Figure 10.6 E) were taken at one meter intervals. Handheld Niton XRF logs were also collected to provide real-time results to guide the drilling and sampling program. Niton analyses were not utilised in the MRE.

#### 10.11.2 Geotechnical Logging

Holes drilled in Kitumba from KITDD_003 (Phase 6) onwards were geotechnically logged. The purpose of the geotechnical logging was to record critical parameters such as weathering, rock strength as well as the type and orientation of important structural features such as joints. The data will be used for stability analysis in mine design and operation, as well as to provide information into particular mining methods such as block caving. Geotechnical logging and sampling of core was carried out according to procedures and guidance supplied by Middindi geotechnical consultants in Phase 6 and Pells Sullivan Meynink (PSM) in Phase 7.

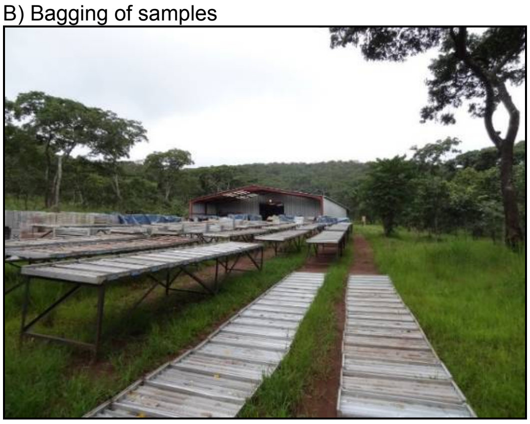
Rock quality designation (RQD) logging was carried out as routine during all phases of drilling at Kitumba. Axial and diametral pointload tests were performed on representative core pieces from the three main geotechnical holes in Phase 7 (KITDD_035, KITDD_036 and KITDD_037).

**Figure 10.6 Core Logging and Sampling at Kitumba**

A) Core on logging stands in the process of being marked up

B) Bagging of samples

C). Core cutting facility



D) Core storage and logging facility

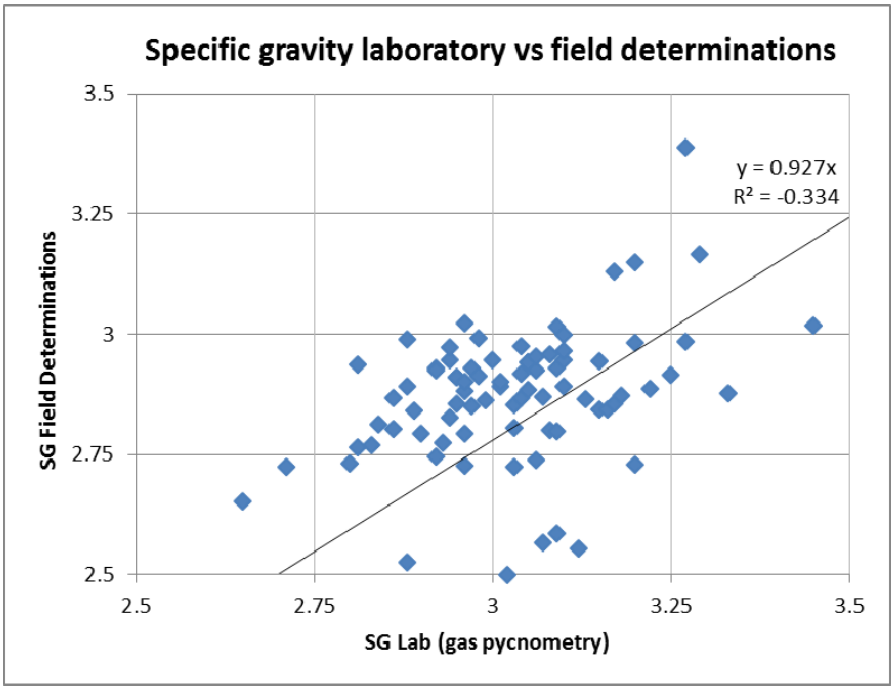
E) Magnetic susceptibililty logging

F) Density measurement using a wet-dry scale

### 10.11.3 Density Determination

Density was determined using a wet-dry scale and the Archimedes principle. Daily calibration checks were carried out (Figure 10.6 F). In addition, 5 percent of Phase 7 hypogene samples were selected for density determination by gas pycnometry at Intertek Genalysis. A reasonably poor correlation is observed between the two methods (Figure 10.7).

**Figure 10.7 Specific Gravity Comparison between Field and Laboratory Determinations for Phase 7 Samples**




## 10.12 Summary of Results

A cut-off grade of 0.25% Cu and a maximum internal dilution of 2 m (drilled width) were used as a guideline when delineating the drilled thickness intervals of mineralisation when reporting length-weighted average grades. True widths were not quoted as the mineralised zone is associated with a sub-vertical tabular zone of brecciation and irregular stockwork-like veining and fracturing. A summary of key intersections within the Kitumba deposit is shown in Table 10.11.

The outline of the Kitumba deposit as illustrated by the 1% Cu cut-off Measured and Indicated Mineral Resource categories is shown in plan view in Figure 10.8 and in cross section and longitudinal section in Figure 10.9 and Figure 10.10 respectively.

The full extent of the Phase 7 Kitumba Mineral Resource area is shown in Figure 10.11 and a series of east-west cross sections through the central high grade core of the deposit in Figure 10.12 to Figure 10.16.

**Table 10.11 Summary of Key Intersections at the Kitumba Deposit (stated at a 0.25% Cu cut-off grade)**

| Hole ID | From | To | Length m | Cu Grade % |
|---|---|---|---|---|
| KITDD_005 | 206 | 426 | 220 | 3.02 |
| KITDD_006 | 55 | 477 | 422 | 1.27 |
| KITDD_006 | 230 | 405 | 175 | 2.03 |
| KITDD_008 | 326 | 522 | 196 | 0.73 |
| KITDD_009 | 203 | 435 | 232 | 1.12 |
| KITDD_010 | 257 | 289 | 32 | 0.87 |
| KITDD_013 | 174 | 393 | 219 | 1.21 |
| KITDD_015 | 103 | 382 | 279 | 0.5 |
| KITDD_016 | 169 | 295 | 126 | 0.53 |
| KITDD_018 | 184 | 218 | 34 | 0.57 |
| KITDD_018 | 297 | 305 | 8 | 1.1 |
| KITDD_019 | 303 | 445 | 142 | 0.49 |
| KITDD_019 | 191 | 261 | 70 | 0.53 |
| KITDD_024 | 304 | 385 | 81 | 1.03 |
| KITDD_025 | 208 | 238 | 30 | 2.02 |
| KITDD_026 | 311 | 422 | 111 | 1.07 |
| KITDD_026 | 213 | 245 | 32 | 1.12 |
| KITDD_027 | 208 | 451 | 243 | 5 |
| KITDD_028 | 240 | 414 | 174 | 5.04 |
| KITDD_029 | 211 | 277 | 66 | 1.58 |
| KITDD_030 | 278 | 462 | 184 | 2.71 |
| KITDD_031 | 199 | 365 | 166 | 7.14 |
| KITDD_032 | 183 | 406 | 223 | 3.22 |
| KITDD_033 | 236 | 346 | 110 | 4.08 |
| KITDD_036 | 321 | 357 | 36 | 2.11 |
| KITDD_037 | 221 | 407 | 186 | 2.25 |
| S36_001 | 170 | 487 | 317 | 0.8 |
| S36_010 | 274 | 458 | 184 | 0.48 |
| S36_014 | 242 | 272 | 30 | 0.5 |
| S36_016 | 142 | 339 | 197 | 0.62 |
| S36_017 | 203 | 431 | 228 | 1.5 |
| S36_023 | 182 | 461 | 279 | 1.1 |
| S36_024 | 142 | 407 | 265 | 0.74 |
| S36_025 | 294 | 529 | 235 | 2.06 |
| S36_026 | 569 | 610 | 41 | 2.31 |
| S36_026 | 277 | 434 | 157 | 0.5 |
| S36_026 | 195 | 202 | 7 | 0.97 |
| S36_029 | 350 | 399 | 49 | 0.57 |
| S36_032 | 311 | 498 | 187 | 2.62 |
| S36_034 | 245 | 464 | 219 | 2.02 |
| S36_035 | 190 | 439 | 249 | 1.33 |
| S36_036 | 198 | 450 | 252 | 1.94 |
| S36_038 | 188 | 493 | 305 | 4.05 |

**Figure 10.8 Kitumba Drill Holes Showing the Footprint of the 1% Cu Cut-Off Measured and Indicated Material.**

**Also Showing the 80 M Cross Section and Long Section Location in Figure 11.4 and Figure 11.5, respectively (Source: Blackthorn)**

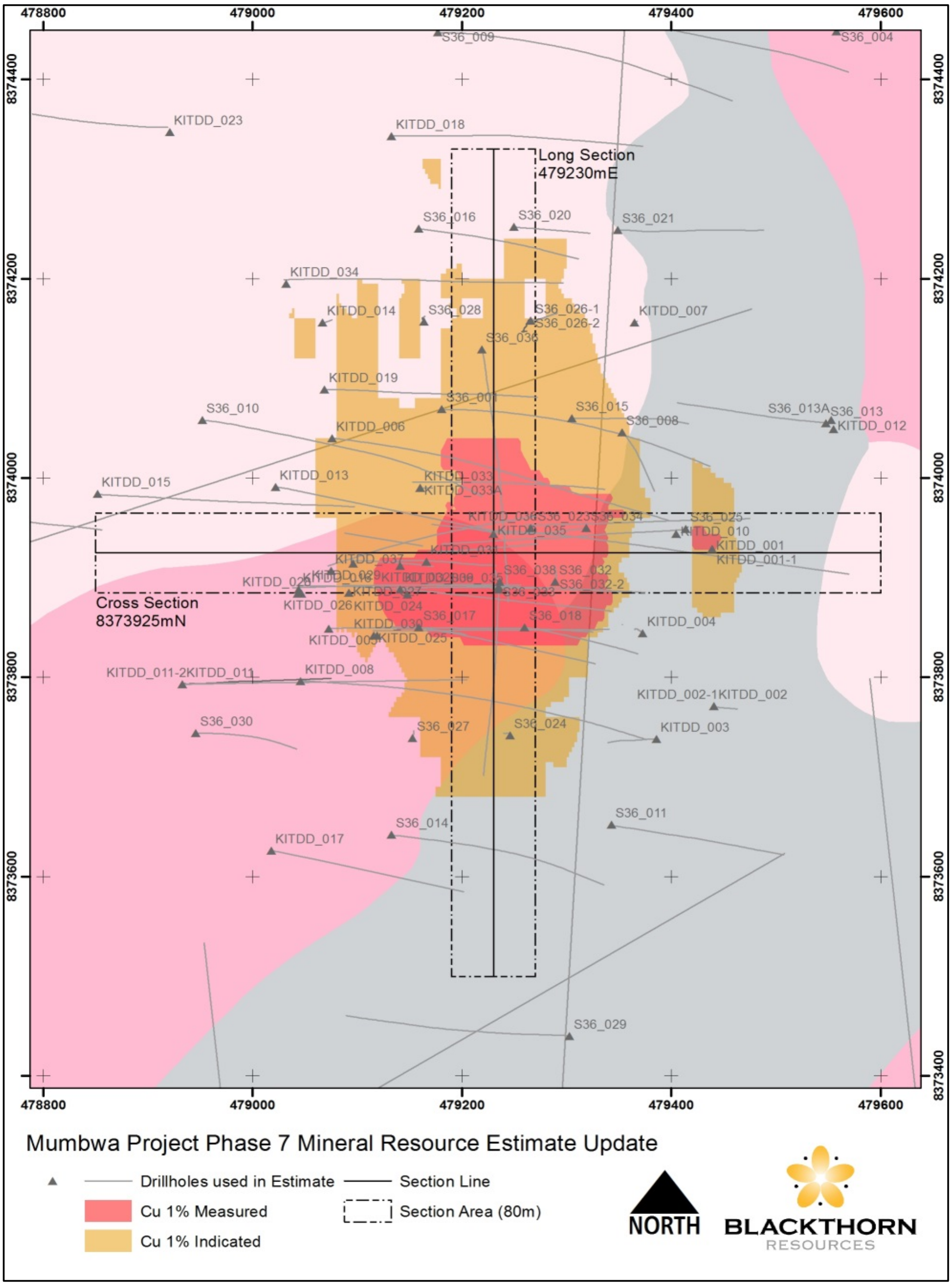
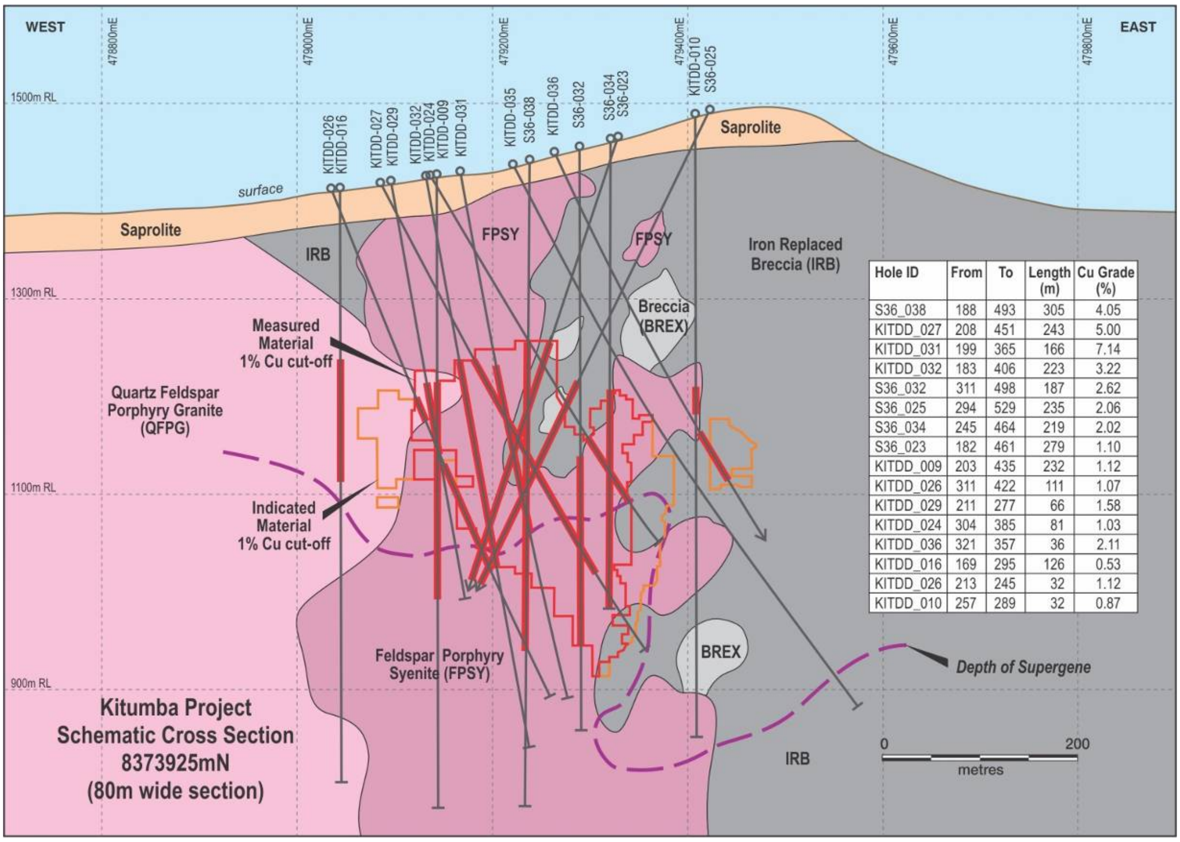
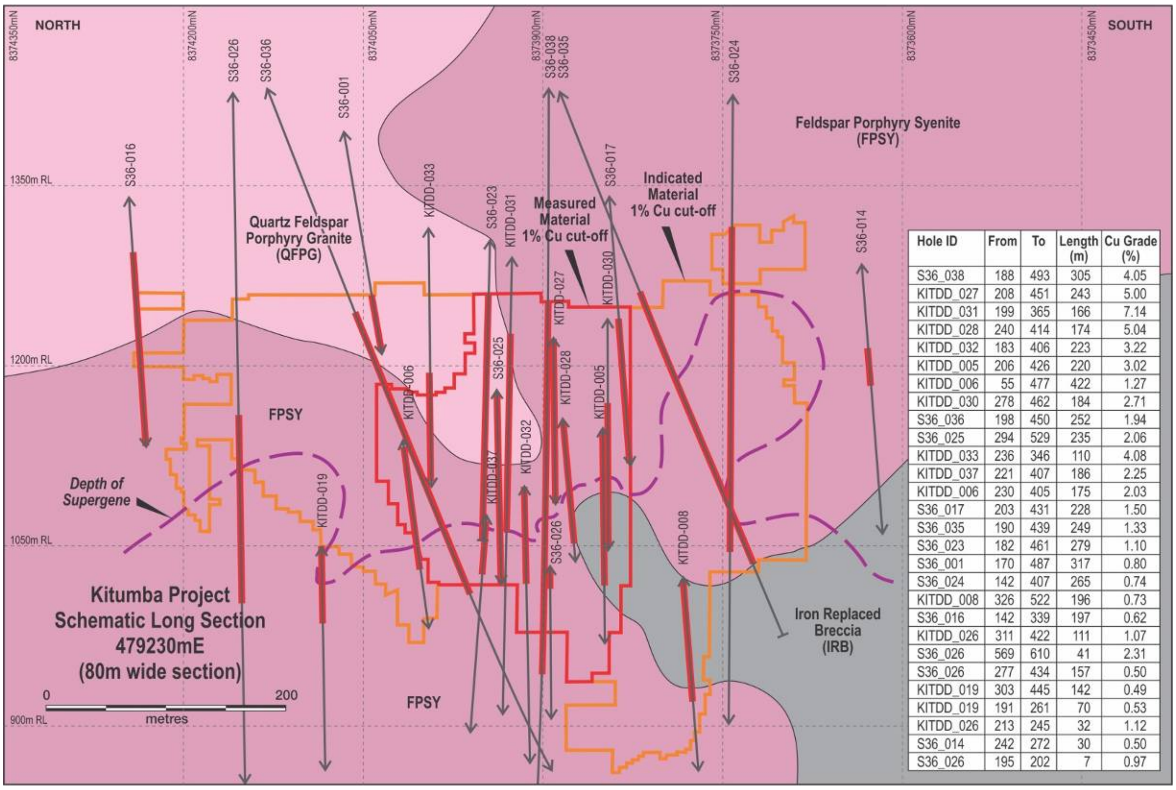


**Figure 10.9 Cross Section 8373925mn, Showing Geology and the Outline of the 1% Cu Cut-Off Measured and Indicated Material (Source: Blackthorn)**




| Hole ID | From | To | Length (m) | Cu Grade (%) |
|---|---|---|---|---|
| S36_038 | 188 | 493 | 305 | 4.05 |
| KITDD_027 | 208 | 451 | 243 | 5.00 |
| KITDD_031 | 199 | 365 | 166 | 7.14 |
| KITDD_032 | 183 | 406 | 223 | 3.22 |
| S36_032 | 311 | 498 | 187 | 2.62 |
| S36_025 | 294 | 529 | 235 | 2.06 |
| S36_034 | 245 | 464 | 219 | 2.02 |
| S36_023 | 182 | 461 | 279 | 1.10 |
| KITDD_009 | 203 | 435 | 232 | 1.12 |
| KITDD_026 | 311 | 422 | 111 | 1.07 |
| KITDD_029 | 211 | 277 | 66 | 1.58 |
| KITDD_024 | 304 | 385 | 81 | 1.03 |
| KITDD_036 | 321 | 357 | 36 | 2.11 |
| KITDD_016 | 169 | 295 | 126 | 0.53 |
| KITDD_026 | 213 | 245 | 32 | 1.12 |
| KITDD_010 | 257 | 289 | 32 | 0.87 |

**Figure 10.10 Long Section 479230me, Showing Geology and the Outline of the 1% Cu Cut-Off Measured and Indicated Material (Source: Blackthorn)**




| Hole ID | From | To | Length (m) | Cu Grade (%) |
|---|---|---|---|---|
| S36_038 | 188 | 493 | 305 | 4.05 |
| KITDD_027 | 208 | 451 | 243 | 5.00 |
| KITDD_031 | 199 | 365 | 166 | 7.14 |
| KITDD_028 | 240 | 414 | 174 | 5.04 |
| KITDD_032 | 183 | 406 | 223 | 3.22 |
| KITDD_005 | 206 | 426 | 220 | 3.02 |
| KITDD_006 | 55 | 477 | 422 | 1.27 |
| KITDD_030 | 278 | 462 | 184 | 2.71 |
| S36_036 | 198 | 450 | 252 | 1.94 |
| S36_025 | 294 | 529 | 235 | 2.06 |
| KITDD_033 | 236 | 346 | 110 | 4.08 |
| KITDD_037 | 221 | 407 | 186 | 2.25 |
| KITDD_006 | 230 | 405 | 175 | 2.03 |
| S36_017 | 203 | 431 | 228 | 1.50 |
| S36_035 | 190 | 439 | 249 | 1.33 |
| S36_023 | 182 | 461 | 279 | 1.10 |
| S36_001 | 170 | 487 | 317 | 0.80 |
| S36_024 | 142 | 407 | 265 | 0.74 |
| KITDD_008 | 326 | 522 | 196 | 0.73 |
| S36_016 | 142 | 339 | 197 | 0.62 |
| KITDD_026 | 311 | 422 | 111 | 1.07 |
| S36_026 | 569 | 610 | 41 | 2.31 |
| S36_026 | 277 | 434 | 157 | 0.50 |
| KITDD_019 | 303 | 445 | 142 | 0.49 |
| KITDD_019 | 191 | 261 | 70 | 0.53 |
| KITDD_026 | 213 | 245 | 32 | 1.12 |
| S36_014 | 242 | 272 | 30 | 0.50 |
| S36_026 | 195 | 202 | 7 | 0.97 |

**Figure 10.11 Plan Showing the Full Extent of the Phase 7 Mineral Resource Area and the Location of a Series of East-West Section Lines through the Central Part of the Kitumba Deposit**

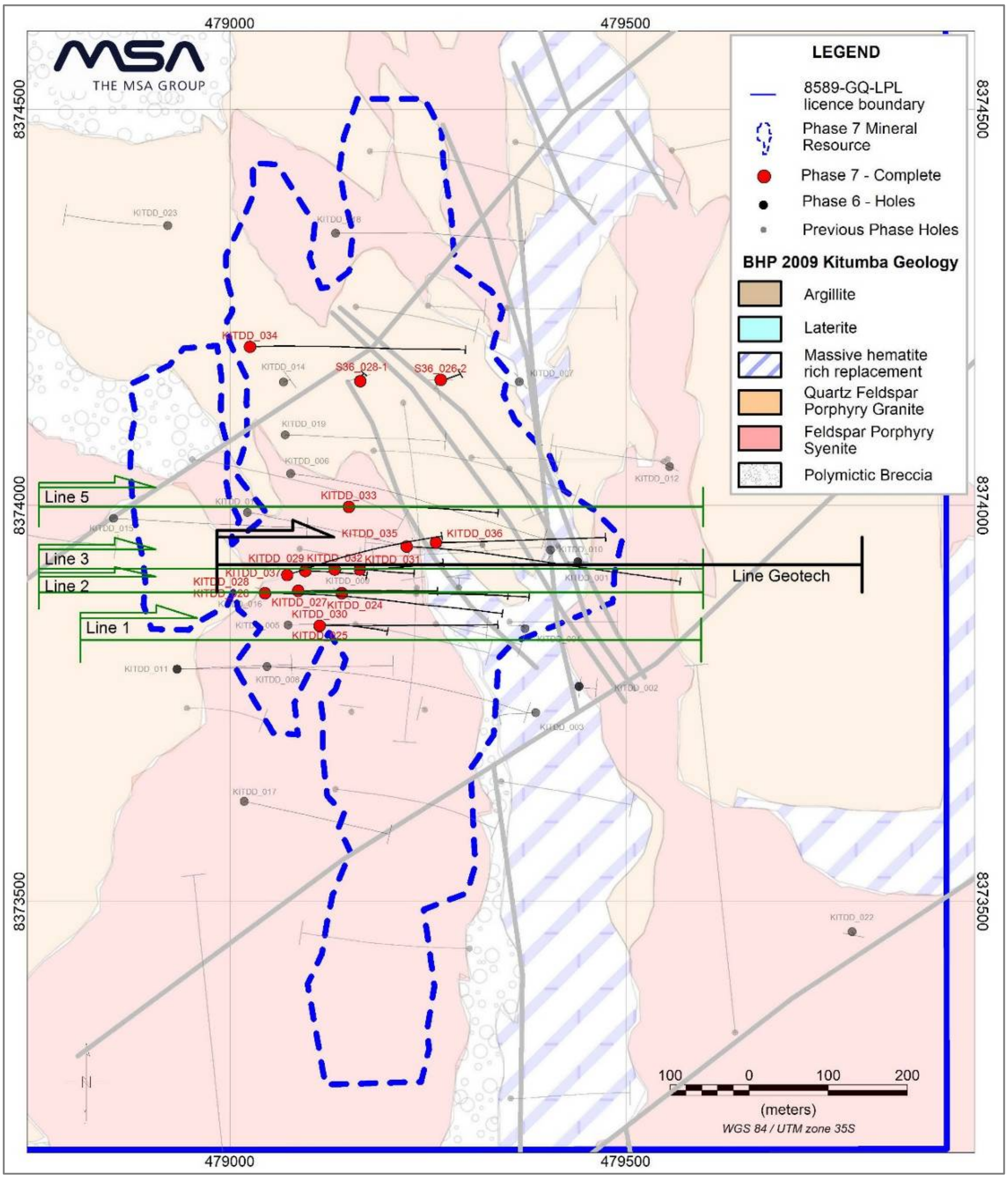


**Figure 10.12 Line 1 Section through the Kitumba Deposit**

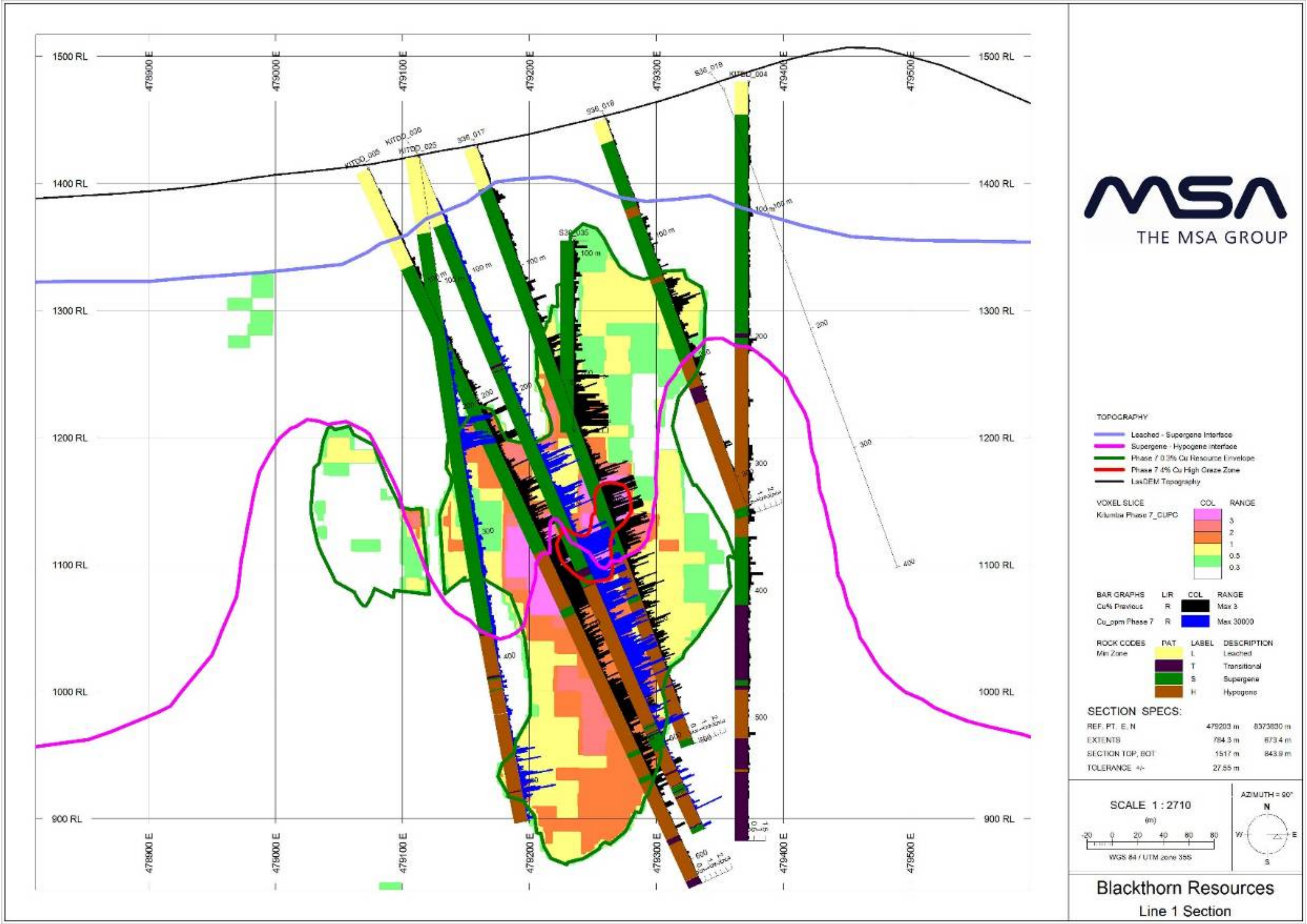


**Figure 10.13 Line 2 Section through the Kitumba Deposit**

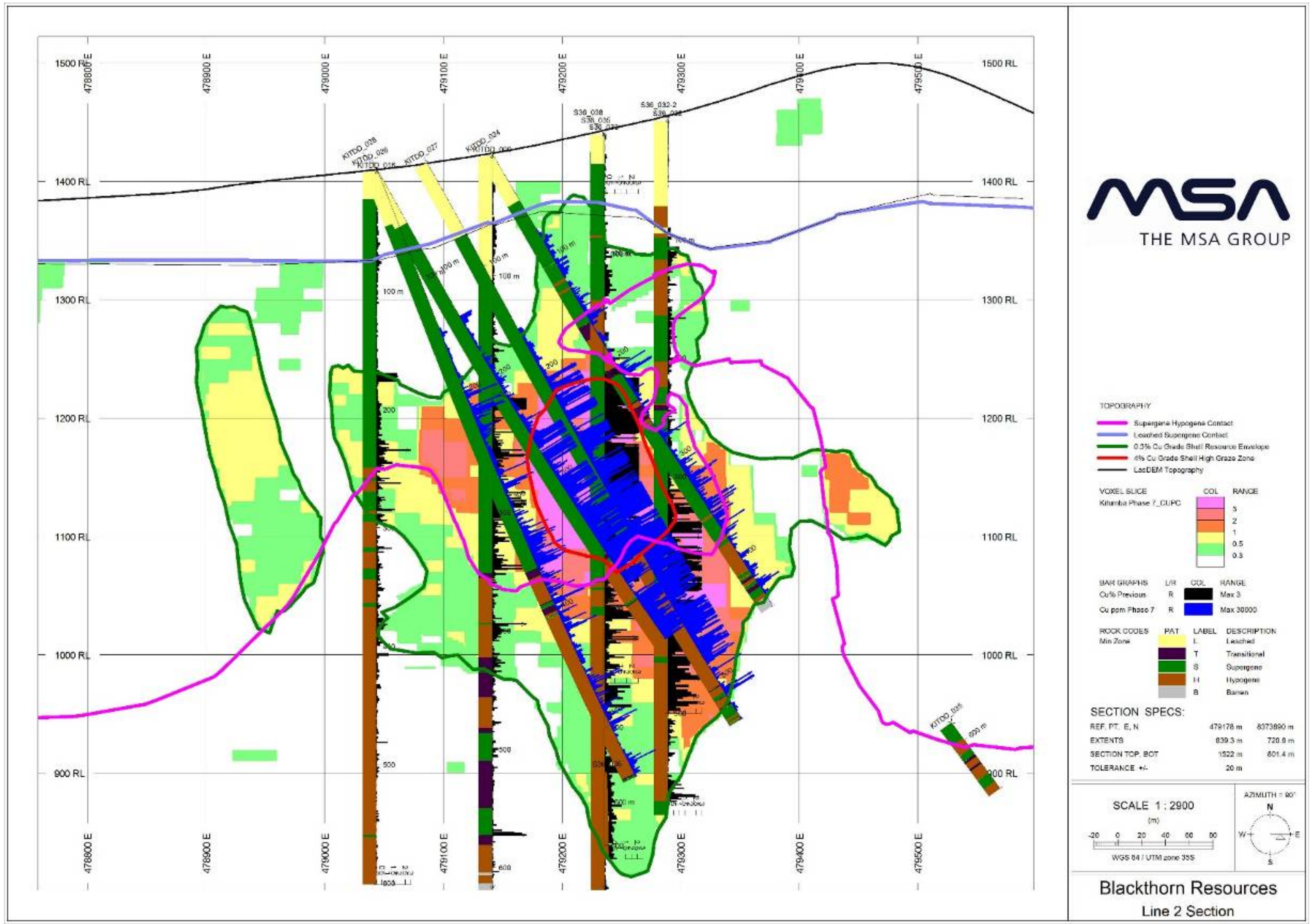


**Figure 10.14 Line 3 Section through the Kitumba Deposit**

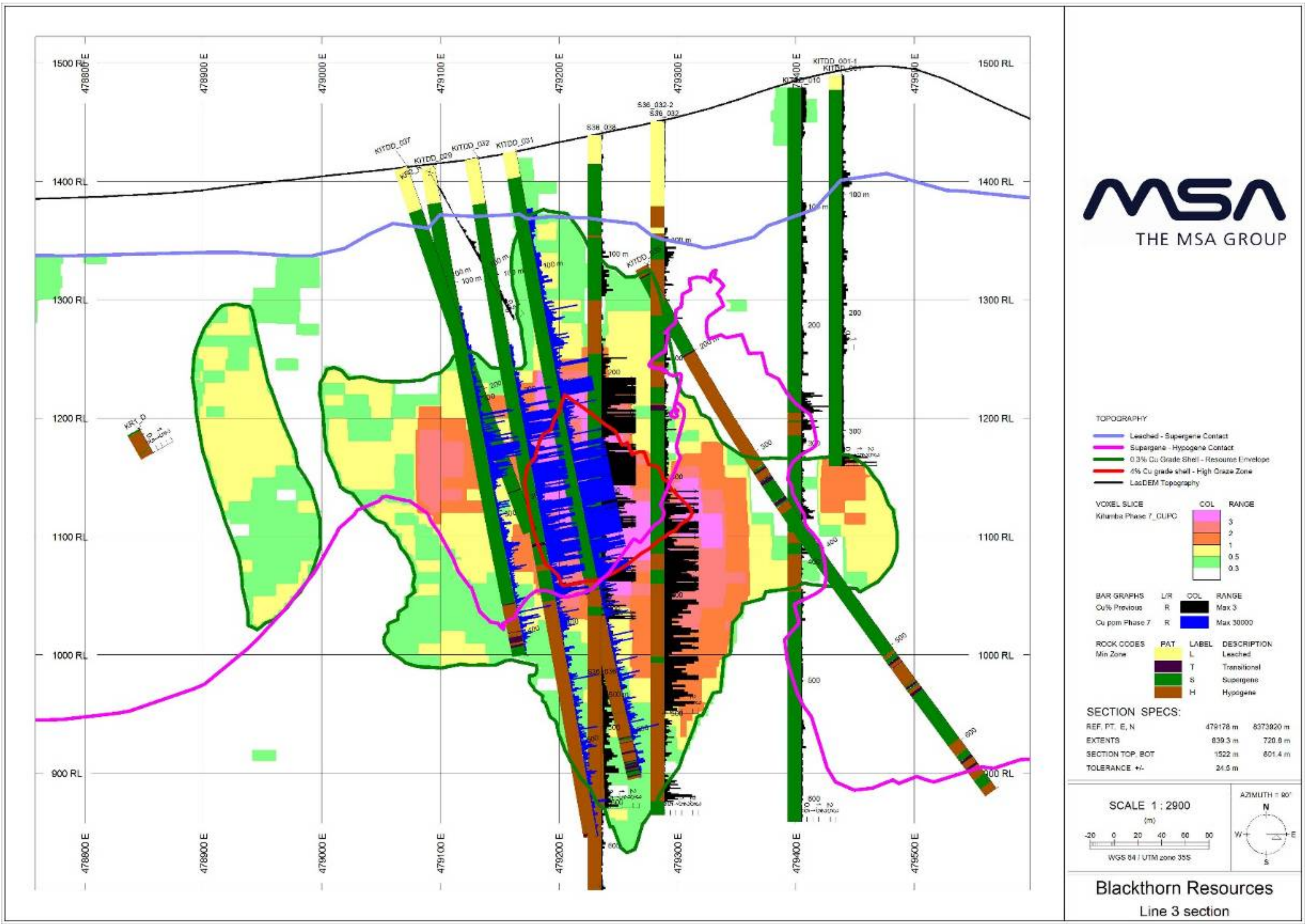


**Figure 10.15 Line 5 Section through the Kitumba Deposit**

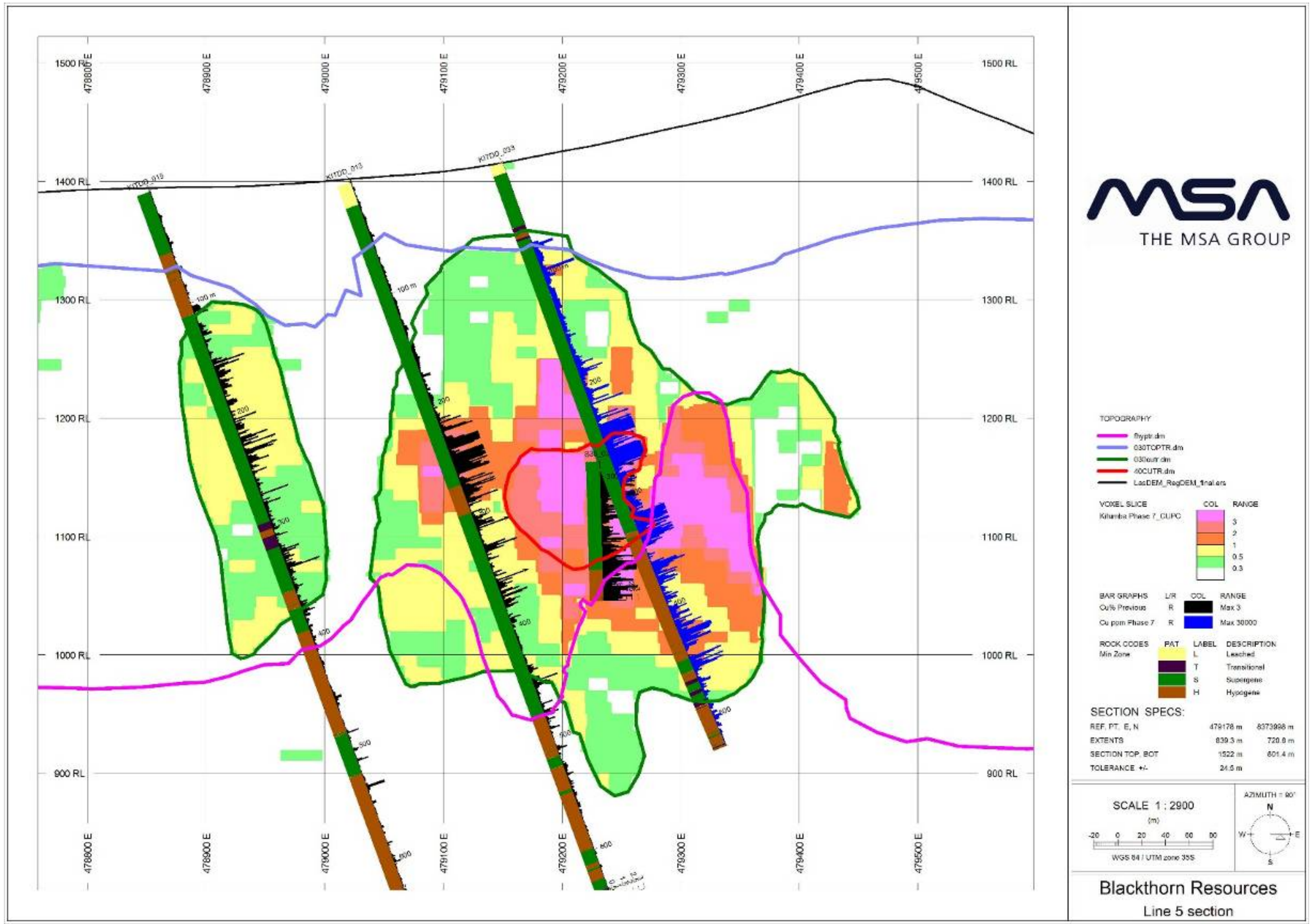
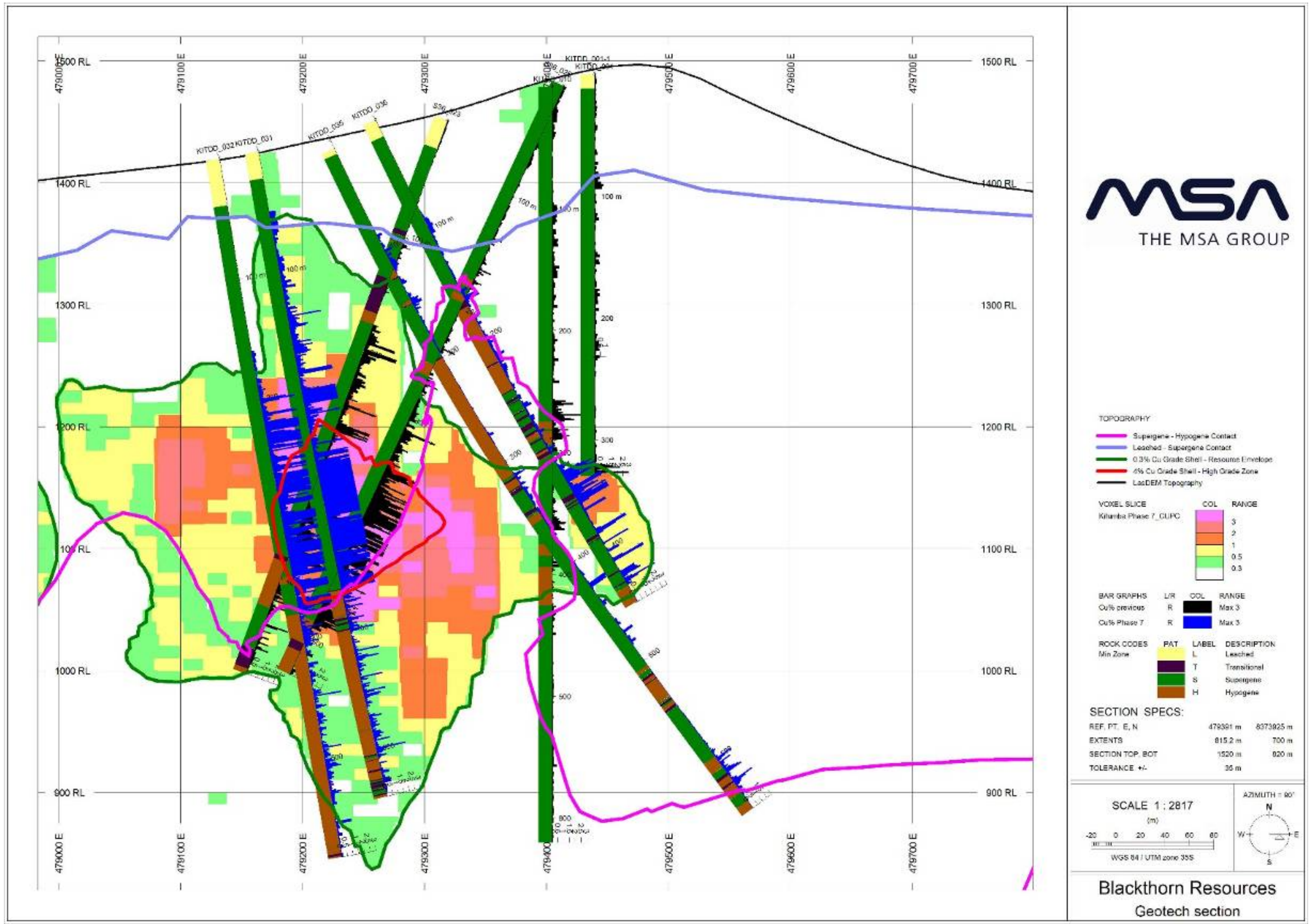


**Figure 10.16 Line Geotech Section through the Kitumba Deposit**



# 11.0 SAMPLING PREPARATION, ANALYSES AND SECURITY

## 11.1 Routine Sampling

The following sampling protocol was adopted at Kitumba:

- Core samples were orientated, marked, cut and sampled according to the SOP (Figure 10.6).
- PQ core was cut in half and cut again to produce quarter core samples; HQ and NQ size core was cut to produce half core samples (with the exception of the metallurgical testwork holes where HQ core was sampled at quarter core).
- Where mineralisation was observed, a sample length of 1 m was used.
- Where no visual mineralisation was present, but alteration (Fe/Mn oxide, hematite, K feldspar, sericite) and/or brecciation were observed, a sample length of 2 m was adopted.
- Outside of these zones, a nominal 2 m sample length was adopted with the exception of the satellite target holes in which no mineralisation was observed. These holes were sampled at 3 m intervals.
- Drill holes were sampled along their entire length.

Core samples were placed in plastic sample bags, sample tickets with unique sample numbers were placed in each bag, along with engraved aluminium tags. The sample bags were stapled closed (Figure 10.6 B) and were placed in large polyweave bags which were then sealed by cable ties, ready for dispatch.

Sample batches comprising numbered and sealed polyweave bags were collected from the Kitumba camp by a contractor and transported by road to the Copperbelt where they were delivered to the sample preparation laboratory.

Samples from the Mumbwa Project have been prepared and analysed at various laboratories through the various phases of exploration. A summary of the laboratory history is given in Table 11.1.

As these laboratories are accredited for the various sample preparation and assay methods used and a similar assay approach was used throughout the history of the project, together with the fact that the majority of the resource assay work was conducted through Intertek Genalysis, the section below describes the procedures as used by Intertek Genalysis.

**Table 11.1 Summary of Preparation and Analytical Laboratories used on the Mumbwa Project**

| Phase | Preparation Laboratory | Assay Laboratory | Assay Methods | Check Laboratory |
|---|---|---|---|---|
| 2 | Genalysis (Johannesburg) | Genalysis (Perth) | Au (fire assay - AAS); Multi-elements (4 acid-ICP) | |
| 3 | Alfred H Knight (Kitwe) | Alaska Assay Laboratories (Fairbanks) | Cu (hot Aqua Regia-AAS); Au (fire assay-AAS) | |
| | | ACME (Vancouver) | Multi-elements (4 acid-ICP) | |
| 3B | SGS (Kalulushi) | SGS (Johannesburg) | Au (fire assay - AAS); Multi-elements (4 acid-ICP) | ALS Chemex (Johannesburg) |
| 4 | SGS (Kalulushi) | SGS (Johannesburg) | Au (fire assay - AAS); Multi-elements (4 acid-ICP) | ALS Chemex (Johannesburg) |
| 5 | Alfred H Knight (Kitwe) | Alfred H Knight (Kitwe) | Total Cu (4 acid-AAS); Acid soluble Cu | Intertek Genalysis (JHB & Perth) |
| | | AHK Geochem (Fairbanks) | Au (fire assay-AAS) | |
| | | ACME (Vancouver) | Multi-elements (4 acid-ICP) | |
| 6 | Intertek Genalysis (Chingola) | Intertek Genalysis (Johannesburg) | Au (fire assay-AAS) | ALS Chemex (Johannesburg) |
| | | Intertek Genalysis (Perth) | Multi-elements (4 acid-ICP) | |
| 7 | Intertek Genalysis (Chingola) | Intertek Genalysis (Johannesburg) | Key elements (Ore grade 4 acid-ICP); Multi-elements (4 acid-ICP) | ALS Chemex (Johannesburg) |
| | | Intertek Genalysis (Perth) | Acid soluble Cu | |

## 11.2 Sample Preparation

On arrival at the preparation laboratory samples were weighed, recorded and reported as received. Any queries or discrepancies were resolved prior to commencement of sample processing. Aspects of the sample preparation process are shown in Figure 11.1.

The entire sample was dried overnight at 105$^0$C (Figure 11.1B) prior to crushing. Samples weighing < 3 kg were crushed to 10 mm nominal particle size and the entire sample was then milled. Samples greater than 3 kg were crushed to 2 mm – 3 mm particle size and a riffle splitter (Figure 11.1 E) used to produce a representative split weighting less than 1.2 kg. Crusher rejects were placed in new barcode labelled bags which were subsequently collected and returned to site. The crusher used was a single stage Essa JC2500 fine jaw crusher (Figure 11.1 D) for 85% passing 2 mm. This is a closed unit and is effective at dust suppression. A screen test was done after every 25 samples, and results documented. Barren rock and compressed air were used for cleaning between samples when adhering material was observed. This material was retained for analysis. MSA inserted quartz blanks within the sample stream after high grade samples to check for cross contamination.

Four Essa LM2 mills were used to mill samples to 85% passing 75µm. Two steel bowls and disc pulverisers were used, one cleaned while the other was in use. A minimum 3 minute milling cycle was used. The disc was routinely weighed to determine loss. Cleaning included a barren quartz flush at the beginning and end of each batch and the material retained for analysis. Cleaning with compressed air and brushing occurred between each sample.

150 g pulp splits were obtained, and stored in barcoded labelled geochemical envelopes (Figure 11.1 C) and exported to the Intertek Genalysis laboratory in Johannesburg, South Africa Figure 11.1 F). The remaining pulp was returned to the sample bag which was subsequently collected and returned to site. On arrival in Johannesburg samples were again checked and sorted.

The QP, Michael Robertson, carried out an audit of the sample preparation facility on the 23 August 2012 and found that appropriate procedures were being adhered to.

## 11.3 Assay Methodology

Sample pulps were digested by a multi-acid mix including hydrofluoric, nitric, perchloric and hydrochloric acids and analysed by Inductively Coupled Plasma Optical (Atomic) Emission Spectrometry (ICP-OES) and Inductively Coupled Plasma Mass Spectrometry (ICP-MS). For samples analysed by ICP-MS the multi-acid digest was conducted in teflon tubes. Over-range assays (by dilution and re-reading) were carried out as routine where the Cu concentration exceeded the upper limit for the method.

During the later phases, samples were derived mainly from the high grade core of the deposit and were analysed by 'ore grade' methods.

Samples were also analysed for acid soluble copper (ASCu) at the Intertek Genalysis Perth laboratory. Acid soluble copper analysis is a sulphuric acid leach that is designed to provide an indication of the amount of acid soluble copper present in a sample. Sulphuric acid will dissolve copper present in oxides or carbonate (e.g. malachite) but will not digest copper present in sulphides. Acid soluble copper methods are empirical i.e. the obtained result is dependent on the conditions under which the digestion took place (time, temperature and acid concentration). These methods are usually variable between laboratories due to slight differences in procedures.

Acid soluble copper was determined by dilute sulphuric acid leach and flame Atomic Absorption Spectrometry (AAS) finish. A 0.5 g aliquot of sample pulp was leached while being agitated continually at a constant set temperature, for one hour, in a 5% $H_2SO_4$ solution. The sample was centrifuged, 1 mL pipetted into a marked tube which was volumed, diluted and analysed on the AAS. The results were corrected for the catch weight by the LIMS system.

The Intertek Genalysis laboratory in Johannesburg laboratory is accredited in accordance with the recognised International Standard ISO/IEC 17025:2005 with Facility Accreditation Number T0464. The accreditation demonstrates technical competency for a defined scope and the operation of a laboratory quality management system. In the context of the Kitumba project, the laboratory is accredited for gold by fire assay and ICP finish, multi-element analysis (ore grade and standard) by 4 acid digestion and ICP-OES and specific gravity analysis by gas pycnometry.

The Intertek Genalysis laboratory in Perth is an accredited NATA (National Association of Testing Authorities, Australia) laboratory (Facility Accreditation Number 3244) and is also accredited in accordance with ISO/IEC 17025:2005.

The Intertek Genalysis laboratories in Chingola and Johannesburg were audited by the QP, Michael Robertson, and found to be compliant with respect to their internal procedures and general industry best practice. Aspects of sample preparation and analysis at the Johannesburg facility are shown in Figure 11.2.

## 11.4 Sample Chain of Custody

An unbroken sample chain of custody was implemented, as follows:

- Sample polyweave bags were sealed with cable ties.
- Sample shipments were examined on arrival at the laboratory and the sample dispatch form signed and returned to Kitumba with a confirmation of the security seals and the presence of all samples comprising each batch.

**Figure 11.1 Sample Preparation at the Intertek Genalysis Sample Preparation Laboratory, Chingola**


A) Signed and stamped sample submission documentation

B) Samples in steel trays in the drying oven


C) Bar coded geochem envelope containing sample pulp

D) Single stage Essa JC2500 fine jaw crusher

E) Stainless steel riffle splitter

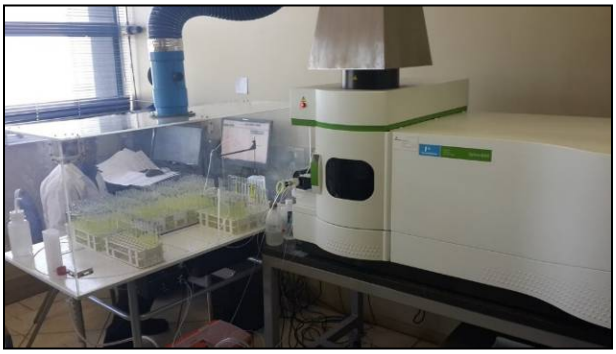
F) Samples packed in boxes ready for export to Johannesburg, South Africa

**Figure 11.2 Analytical Procedures at the Intertek Genalysis Laboratory, Johannesburg**

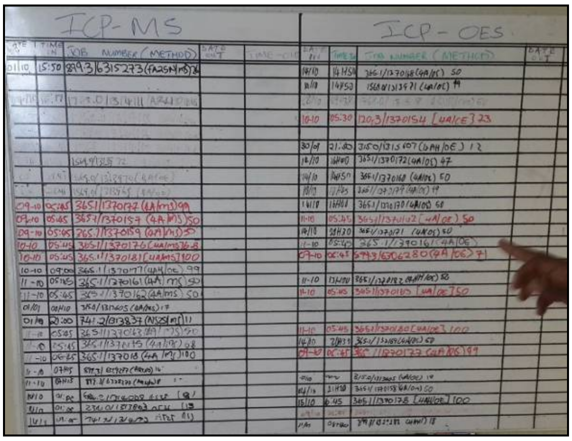
A) Pulp storage facility

B) Fire assay laboratory

E) Ore grade 4 acid digest



D) ICP-OES finish on ore grade digests




E) ICP schedule by job number (the majority of this work was for Blackthorn Resources

F) AAS finish on gold fire assay analyses

## 11.5 Quality Control Procedures

A quality assurance and quality control (QAQC) protocol was adopted for the project, and formed part of the SOP.

QAQC on sampling and assaying was achieved by monitoring four essential components:

- Field sample collection.
- Laboratory sample preparation and sub-sampling.
- Analytical accuracy and precision.
- Reporting (clerical or data transfer) accuracy.

### 11.5.1 Laboratory Internal QC

An audit of the Chingola preparation laboratory was undertaken by the QP, Michael Robertson, and procedures were found to be generally acceptable in terms of laboratory best practice. Intertek Genalysis holds NATA ISO/IEC 17025 accreditation.

Standard internal QC protocol at the Intertek Genalysis Group includes a requirement that a minimum 15% of results reported represent QC analyses, including sample repeats, certified standards and blanks.

### 11.5.2 External QC

The external QC program managed by MSA comprised the use of Certified Reference Materials (CRMs) or standards, certified coarse and pulp blank samples, and field (coarse) and pulp duplicates. The CRMs, blanks and field duplicates were inserted at the Kitumba camp into the stream of core samples comprising each batch. Although these were distinguishable as QC samples, they followed the sequential numbering of the batch and their identity was not revealed to the laboratory. External QC samples comprised 15 % of all sample submissions.

### 11.5.3 Certified Reference Materials (Standard Samples)

The use of CRMs provides a measure of analytical accuracy, and is useful in identifying any short or long-term drift and biases. Longer term problems are ideally identified by a ‘drift’ in the actual standard values over time or by a significant number of points that congregate either above or below the certified mean.

CRMs should be similar in chemical composition (matrix matched) to the exploration samples, have a certified mean value and associated confidence limits, and be certified for the analytical method being used. The CRMs used were selected to cover the expected grade range and deposit type.

Through the monitoring of the expected value of the CRM within a sample batch of unknown and variable values, the accuracy of the assay process was quantified. Monitoring was undertaken using control charts where the results for each CRM were plotted on a time sequential basis. These charts are presented in Appendix 4.

Potential problems in analytical accuracy were identified by any result lying outside two standard deviations limits about the certified mean. Such problem values were initially checked to determine if they related to transcription (both field and laboratory) errors.

### 11.5.4 Blank Samples

Blank samples which have been established as barren by previous analysis, serve to detect contamination during the sample preparation and assay process and to act as a 'wash' between and after the analysis of known high grade samples. They are also useful for monitoring mis-sequencing of samples during sample preparation and analysis.

Two certified blank samples, AMIS0108 and AMIS0305, were used, both being sourced from African Mineral Standards. AMIS108 is a silica pulp and AMIS305 comprises silica chips. Blanks were inserted at the beginning of each batch to monitor between-batch contamination and internally within and immediately following mineralised intervals in order to monitor within-batch contamination.

The performance of the blanks was monitored by control charts. Warning limits were set at five times the detection limit, being 0.05 % for copper and 0.005 ppm for gold. Samples returning below detection were assigned half the detection limit as a standard approach in the database. Several coarse blank failures are observed, with a maximum 0.20% Cu which indicates poor cleaning on occasions, particularly between and after high grade samples.

### 11.5.5 Duplicate Samples

Duplicate samples can be created at any point in the sampling and sub-sampling process where the mass of the sample is being reduced (i.e. the sample is split and a reject is created). Duplicate samples are used in determining the quality of sample collection, sample preparation and analytical precision. Comparison of pulp duplicate and field duplicate data can indicate where imprecision in the data originates from, i.e. sample preparation versus assay process. The inclusion of duplicate samples and their comparative analysis is essential for determining the level of precision, and hence repeatability, of the analytical method.

Field (coarse) duplicate sample results for total copper are included in Appendix 4. Correlation for copper duplicates is poor, despite a correlation coefficient of 0.94 with a relatively wide scatter and a number of outliers. These results are however expected, owing to the nuggety concentrations of secondary copper minerals such as malachite and chalcocite within the same sample. The correlation between pulp duplicate pairs is significantly better as expected, with a correlation coefficient of 0.996.

### 11.5.6 Failure Criteria

The results of QC samples were used to accept or reject the results of sample batches analysed by the laboratories. Laboratory results were reviewed on a batch by batch basis. In order to accept / reject samples or batches, it was necessary to establish a set of failure criteria. All failures were recorded, identifying the reason for failure and corrective action required. This information was updated once the results of corrective action were received.

### 11.5.7 Umpire Programme

In compliance with standard practice, 5% of sample pulps from Phase 3B onwards were randomly selected across the grade range, retrieved from the primary laboratory, and submitted to a second accredited laboratory for check analyses. ALS Chemex in Johannesburg has been used as the check laboratory for much of the samples derived from resource drilling, particularly from the high grade core of the deposit.

The results show an acceptable correlation between the original and check / umpire assay results with only a few outliers at a grade of >1% Cu (Appendix 4). As expected, more scatter is observed in the acid soluble copper results, likely the result of slight variations in digestion procedure between the primary and check laboratories. Acid soluble copper methods are generally considered empirical in nature i.e. the obtained result is dependent on the conditions under which the digestion took place (time, temperature and acid concentration). Any variation from the method gives varying results and therefore method consistency is paramount.

No material biases are observed and the check umpire results confirm the use of the primary assays in Mineral Resource estimation.

## 11.6 Database

Initially (Mid Phase 6 and earlier) field data were captured in MS Excel templates that were setup with built in validations, and later the data were captured into Logchief™™ which is the data capture application used on site. Logchief™™ is designed to simplify field data collection and management. Logchief™™ utilises the validations and look-ups setup in Datashed to deliver accurate and clean data to the Datashed database that resides on MSA's servers in Johannesburg. The data in Logchief™ is directly synchronised into the master database to ensure that data is backed up and saved to the master database regularly. The database was designed with primary keys and foreign key (Field Lookups) setups. Listed below (Table 11.2) are the tables in the database.

**Table 11.2 Tables in the Phase 7 Database**

| Table | Records |
|---|---|
| Collar | 21 |
| Lithology | 1,212 |
| Alteration | 1,590 |
| Mineralisation | 2,957 |
| Sample and Assay | 5,719 |
| Survey | 1,864 |
| Geotech | 4,921 |
| Density | 5,736 |
| Duplicates | 2,335 |
| Core Photography | 2,605 |
| Core Recovery | 3,640 |
| Metadata | 72 |
| Standards | 422 |

Data were validated at regular intervals as the data were imported, as well as before generating QAQC reports and modelling exports. A validation report was generated for each validation cycle and corrections were made to the database.

## 11.7 Conclusion

The QP considers the sample preparation, security and analytical procedures employed to be appropriate and adequate for an exploration program of this nature. No aspect of the sample preparation or analysis was conducted by an employee, officer, director or associate of Blackthorn. Sufficient reference materials were used to control analytical processes, appropriate analytical procedures were used that take rock matrices into account and provide acceptable levels of precision, and sufficient checking work was carried out to demonstrate that the data are unbiased and acceptable for use in geological modelling and Mineral Resource estimation.

# 12.0 DATA VERIFICATION

A number of data verification and QAQC procedures have been applied during drilling, sampling, assay and data management through the various phases of exploration, in order to ensure the veracity of the data.

The MSA Group and the QP, Michael Robertson, have been involved in the Mumbwa project since 2006 (pre-discovery), and apart from Phases 3B and four managed by BHP Billiton, have been responsible for turnkey exploration management of the project. This has included:

- Development of project-specific standard operating procedures (SOPs) covering all aspects of the exploration work programme and data management.
- Validation of all data on import into the MSA-managed central project database.
- QAQC of assay results.
- Verification of mineralised intersections and Competent Person sign-off of News Releases under the JORC Codes (2004 and 2012 Editions) on behalf of Blackthorn Resources Limited.
- Validation of all input data into the various Mineral Resource estimates undertaken by MSA.
- Numerous site visits by the QP between 2006 and 2014.

The various phases were staffed by different geologists which resulted in slight differences in understanding and logging approach. As a result of improved understanding of the Kitumba mineralised system, a total of 30,322 m was relogged for lithology, alteration, mineralisation and mineralisation zone in order to conform to the revised logging protocols and ensure standardisation in the database used in the Mineral Resource estimate following Phase 6. These procedures continued to be applied during the subsequent Phase 7.

Based on the data verification procedures applied over the history of the project, it is the QP's opinion that the quality of the drilling, sampling and assay data are considered to meet or exceed the standards required by JORC and NI 43-101 and that the data are suitable for use in Mineral Resource estimation.

# 13.0 MINERALS PROCESSING AND METALLURGICAL TESTING

## 13.1 Introduction

Testwork undertaken for the September 2013 PFS focussed on production of a saleable flotation concentrate and recovery of acid soluble copper from the flotation tails with an atmospheric tank leach. Copper recoveries were however, lower than desired and operating costs were high due to the requirement to truck large quantities of sulphuric acid from the Copperbelt smelters.

It was recognised that the copper mineralogy of the Kitumba deposit is complex and that a different approach to that used in the PFS was needed to maximise copper recovery and to off-set the importation of sulphuric acid, a major contributing factor to the operating costs.

As a consequence, an alternative flowsheet was proposed and an attendant testwork programme developed to validate the revised flowsheet proposal. The testwork programme evaluated sulphide rougher flotation followed by pressure oxidation (POX) leaching (autoclaving) of the sulphide concentrate and hot acid ferric leaching of the rougher tailings. Copper recovery from the combined POX and acid ferric leach slurries will be by solid-liquid separation followed by solvent extraction and electrowinning (SX/EW).

The testwork comprised two major programmes of work to demonstrate the viability of the proposed commercial flowsheet, while a third programme to investigate the amenability of the Kitumba ore to heap leaching was also completed. The first was undertaken on a sample designated Composite 1, representing predominantly supergene material from the early years of the life of mine. This was followed by a second batch testwork programme using a sample designated Composite 2 from drill core deemed more representative of Years 5 onwards and thus likely to contain a significantly lower proportion of secondary copper minerals and a commensurately higher proportion of primary sulphide material. The third programme was performed on Composite 3, blended from selected drill core deemed representative of the initial years of the life of mine and thus predominantly Supergene ore.

The testwork results and interpretation are discussed below.

## 13.2 Composite 1 Testwork Summary

The purpose of the Composite 1 batch testwork programme was to supplement the existing testwork database for a different flowsheet to that proposed in the earlier PFS and to provide sufficient data on unit operations to allow the revised conceptual plant design to proceed.

The Composite 1 batch testwork included the following:

- Composite 1 head assay and mineralogy.
- Flotation (rougher flotation of ore and phosphate flotation of rougher tails).
- Atmospheric acid ferric leaching of the rougher tails.

- Solid-liquid separation tests (settling rate, flocculant screening) on the atmospheric leach discharge slurry.

The Composite 1 batch testwork results are summarised as follows:

- The head grade of Composite 1 contains 2.44% Cu and 0.26% S (total).
- The main copper bearing minerals in the ore were identified as chalcosiderite (5.2%), brochantite (1.8%), malachite (2.4%) and amorphous minerals (27.4%).
- The ore has low water soluble copper (0.002% of Cu is water soluble) and sulphur (0.73% of S is water soluble).
- The rougher flotation results show a four times copper grade upgrade, 4.8% mass recovery and 19% Cu recovery in the rougher concentrate. However, sulphur grade in the rougher concentrate is low (2.38% S) and this makes the concentrate unsuitable for POX testing.
- The following are the observations from the rougher tails atmospheric leach testwork results:
  - The copper extraction improved slightly at a finer grind size (Leach 4 at 83.7% Cu extraction with 75 µm grind size compared to Leach 2 at 82.3% Cu extraction with 150 µm grind size).
  - The copper extraction improved slightly at higher leach temperature (Leach 4 at 83.7% Cu extraction with 80°C leach temperature compared to Leach 3 at 79.5% Cu extraction with 60°C leach temperature).
  - The best copper extraction was achieved in Leach 5 (60°C, $P_{80}$ 150 µm, 10 h residence time – 6 h oxidative followed by 4 h reductive leach with $SO_2$ gas) with 86.2% Cu extraction.
  - The solution kinetic assays show that, in all five leach tests, most of the copper is extracted within the first hour of leaching.
- The leach residue mineralogy indicates a significant amount of the copper bearing mineral chalcosiderite remaining in the leach residue. Copper bearing brochantite and malachite minerals appeared to have been fully solubilised during the leaching process. The presence of chalcosiderite in the leach residue prompted investigation into the deployment of a separate phosphate flotation step on the sulphide flotation tail. It was envisaged that phosphate flotation may be able to recover the phosphate mineral (chalcosiderite) to the phosphate flotation concentrate so that it may be combined with the sulphide flotation concentrate and subsequently treated via the POX leaching circuit.
- The phosphate flotation results to date show poor recovery of phosphate to the flotation concentrate.

- The flocculant screening testwork showed that, from the tests that produced clear overflow, Magnafloc 351 achieved the best settling rate at 5.422 m/h at a flocculant addition rate of 24.4 g/t.

- The leach residue thickens relatively poorly (achieved thickening flux of 0.2 $m^2$ /t.day with 45% $^w/_w$ underflow density at 48.9 g/t flocculant addition. These poor thickening results could be due to a significant amount of amorphous minerals present in the leach residue mineralogy which could contain clay minerals. Clay minerals have a tendency to swell causing difficulties in thickening.

- Pressure oxidation testwork was not performed on Composite 1 material as the rougher flotation concentrate produced contained insufficient sulphur to sustain auto-thermal autoclave operation.

- The tailings neutralisation testwork was not performed on Composite 1 material as the leach slurry had aged due to the long period between testwork (due to the laboratory shutdown during the Christmas period).

The early leaching testwork (Leach 1) on the Composite 1 flotation tailings provided indicative Cu recoveries of c.80%. The ability to achieve a higher Cu extraction appears to be hampered by lack of primary sulphides in this Composite material. A decision was made to compile a second Composite (Composite 2) sample from the drill core which is more representative of Years 5 onwards and likely to contain a significantly lower proportion of secondary copper minerals. This formed the basis of the subsequent Composite 2 metallurgical testwork programme.

## 13.3 Composite 2 Testwork Summary

The purpose of the Composite 2 batch testwork programme was to supplement Composite 1 testwork (Year 1 to 5) with material more representative of Years 5 onwards and likely to contain a significantly higher proportion of primary copper minerals.

The Composite 2 batch testwork included the following:

- Blending of Composite 2 core samples to obtain a head Composite sample of c. 47.3 kg.

- Composite 2 head assay and mineralogy.

- Milling (to a $P_{80}$ of 106, 150, and 180 µm) and subsequent flotation of a 10 kg split of the Composite 2 material to produce rougher concentrate and rougher tails samples.

- POX leaching of rougher flotation concentrate.

- Atmospheric acid ferric leaching of the rougher tails.

- Solid-liquid separation tests (settling rate, flocculant screening) on the atmospheric leach discharge slurry.

- Tailings neutralisation testwork on the washed atmospheric leach discharge residue.

The Composite 2 batch testwork results are summarised as follows:

- The head grade of Composite 2 contains 3.54% Cu and 2.54% S (total).

- The ore has low water soluble copper (0.0003% of Cu is water soluble).

- The rougher flotation testwork shows a reasonable copper upgrade (5 to 6 times) and sulphur upgrade (7.5 to 8.5 times) which translates to a high recovery of copper (63 - 69%) and sulphur (95 - 97%) to the rougher concentrate. The results also show that the copper recovery to the concentrate increases with finer grind size.

- The following are the observations from the atmospheric leach testwork results:

  - Cu extraction is largely unaffected by grind size (at the grind sizes tested).

  - The highest Cu extraction was achieved in Leach 12 (60°C, $P_{80}$ 150 µm, 10 h residence time – 6 h oxidative followed by 4 h reductive leach with $SO_2$ gas) with 94.6% Cu extraction. The Cu extraction increased following the addition of sulphur dioxide to reduce the Eh from 536 mV to approximately 400 mV.

  - The solution kinetic assays show that, in all four leach tests, the majority of the Cu is extracted within the first hour of leaching.

  - The extraction from the rougher tails atmospheric leach for Composite 2 (Leach 7 with 84.2% Cu extraction and 165 kg/dry t $H_2SO_4$) shows a higher copper extraction but at higher acid addition compared to the equivalent Composite 1 leach test (Leach 1 with 78.3% Cu extraction and 76 kg/dry t $H_2SO_4$).

- The results of the POX leach tests performed on the rougher concentrate under the prescribed conditions indicate high extraction of copper (>98% Cu extraction) with a solid mass loss of 34 - 38% and calculated POX discharge free acid of 40 - 49 g/L.

- The overall leach copper extraction values were calculated from the rougher tails atmospheric leach and the rougher concentrate POX leach testwork results. The results show that grind size does not affect the overall copper extraction (93.2 - 93.9% overall Cu extraction). The oxidative-reductive rougher tails atmospheric leach shows a 3% copper extraction increase (96.8% overall Cu extraction).

- The flocculant screening testwork results are as follows:

  - For Leach 6 residue ($P_{80}$ 106 µm), Magnafloc E 10 achieved the best settling rate at 10.84 m/h with a flocculant addition rate of 45 g/t.

- For Leach 7 residue ($P_{80}$ 150 µm), Magnafloc 351 achieved the best settling rate at 12.19 m/h with a flocculant addition rate of 30 to 40 g/t.

- For Leach 8 residue ($P_{80}$ 180 µm), Magnafloc 351 achieved the best settling rate at 9.76 m/h with a flocculant addition rate of 40 g/t.

- Overall, the best settling rate was achieved with Leach 7 residue with the addition of 30 g/t to 40 g/t of Magnafloc 351.

• The leach residue thickens relatively poorly (achieved thickening flux of 0.36 $m^2$/t.day with 55% $^w/_w$ underflow density at 40 g/t flocculant addition) – the poor thickening results may be due to the presence of amorphous minerals in the ore. The leach residue may contain clay minerals which have a tendency to swell and thus adversely affect solid-liquid separation efficiency.

• The tailings neutralisation results show that limestone was not required as the initial feed slurries were already at a pH higher than the limestone target pH of 4. However, lime will be required at a consumption rate of around 2 to 3 kg/t.

## 13.4 Composite 3 Testwork Summary

The purpose of the Composite 3 batch testwork programme was to determine the amenability of the Kitumba ore to heap leaching and thus assess whether heap leaching would provide a more cost effective flowsheet option compared to pressure oxidation and atmospheric leaching.

The Composite 3 batch testwork included the following:

• Composite 3 head assay.

• Crushing to 3 separate size fractions (-6.73 mm, -12.5 mm, -25.4 mm).

• A separate bottle roll leach test performed on each size fraction.

A summary of the Composite 3 bottle roll test results is as follows:

• The head grade of Composite 3 contained 3.42% Cu.

• Bottle roll leach tests performed on three different crushed ore size fractions yielded low copper extraction over the seven day leaching period. The highest copper extraction of 36% (solids basis) was achieved for the finer size fraction (-6.73 mm), while a negative copper extraction (-9%) was returned for the coarse fraction (-25.4 mm) indicating the presence of native copper.

• The bottle roll sighter leach tests indicated that, although a portion of the copper was leached, the low overall copper recovery would not justify heap leaching as a viable alternative to POX / atmospheric acid leaching.

# 14.0 MINERAL RESOURCE ESTIMATES

## 14.1 Introduction

MSA has completed a Mineral Resource Estimate for the Kitumba deposit on behalf of Blackthorn. Since the previous Mineral Resource Estimate was completed by MSA in April 2013, the Phase 7 infill drilling programme has been completed which provided a significant amount of additional diamond drilling data designed to enhance the confidence of the central high grade portion (core) of the Mineral Resource.

MSA has been involved with the Mumbwa project since the Phase 2 drilling campaign in 2006 and is familiar with all aspects of the project. The Qualified Person for the Mineral Resource estimate, Mr J.C. Witley, visited the project in November 2012 in order to familiarise himself with the mineralisation at Kitumba and again in August 2013 in order to review the core from the Phase 7 drilling programme.

The Mineral Resource Estimate was conducted using CAE Studio 3 (Datamine) software, together with Microsoft Excel and Snowden Supervisor for data analysis.

The principal sources of information used for the estimate include raw data generated during the course of a number of drilling programmes including the most recently completed phase of infill drilling (Phase 7). Infill drilling has taken place since the effective date of this Mineral Resource (5 December 2013), however the results are unlikely to materially affect the Mineral Resources reported herein.

## 14.2 Mineral Resource Estimation Data

The data that directly informs the Mineral Resource Estimate consists of:

- Information from diamond drill cored drillholes:
  - Collar surveys
  - Down-the-hole-surveys
  - Sampling and Assay data
  - Specific Gravity measurements
  - Geology logs including mineralisation state
- A topographic surface
- Various geological interpretations that were conducted using Leapfrog software by the BRL geologists.

The drillholes were collared at various orientations, from inclined at various angles and directions through to vertical. Until Phase 7, the drillholes covered a grid between approximately 80 m and 100 m apart in the east-west direction along fence lines spaced between approximately 80 m and 100 m apart in the north south direction, although peripheral areas were less well drilled, up to 200 m apart. The infill drilling provided a drilling grid of between 20 m and 40 m covering an irregular area in the order of approximately 200 m north to south by 160 m east to west. The infill drilling covered depths from surface down to approximately between 500 m and 600 m below surface. The different dips and directions of the drillholes have created some areas where drillholes overlap which, together with one high grade intersection that was twin-drilled (S36-033 and S36-038), provided useful information on short range continuity of the mineralisation.

The data cut-off date for inclusion of data into this Mineral Resource estimate was 5 November 2013.

## 14.3 Exploratory Statistical Analysis of the Raw Data

The drillhole dataset consists of sample and logging data from diamond drilled (DD) drillholes and reverse circulation (RC) drillholes. Although the samples were assayed for a number of elements, only the following attributes were considered directly relevant to this Mineral Resource estimate:

- Total Copper in percent (CUPC).
- Acid Soluble Copper in percent (CUASPC).
- Cobalt in parts per million (COPPM).
- Gold in grams per tonne (AUGT).
- Silver in grams per tonne (AGGT).
- Uranium in parts per million (UPPM).
- Iron in percent (FEPC).
- Sulphur in parts per million (SPPM).
- Manganese in parts per million (MNPPM).
- Relative Density (SG).

The modelling field names are shown in brackets, these field names being used extensively in this report.

Two fields occur in the database for copper. The first has an upper limit of 5% and the second contains the over-limit assays. Where an over-limit assay occurred, the original assay was replaced with the over-limit value. A number of upper-detection limits (UDL's) were in-use for manganese during the various phases of drilling. For the S36 series the manganese UDL was

10,000 ppm, for the KITDD_016 to 032 holes it was 20,000 ppm, and 50,000 ppm for KITDD_001 to 014. Only samples from holes newer than KITDD_023 were subject to over-limit assays and consequently manganese will be understated in the database. This is not considered material to this Mineral Resource estimate.

Not all of the samples were assayed for the elements of interest, particularly during the earlier drilling campaigns. All of the samples that were assayed were assayed for CUPC. Should CUPC not be assayed it was assumed that there was no visible mineralisation and the CUPC, CUASPC, UPPM, AUGT, AGGT and COPPM values were set to zero. FEPC and MNPPM were left as null values as rocks barren in copper will still contain significant iron and manganese. The null value allows the estimation process to create an estimate using surrounding actual values rather than assigning a zero grade.

Visual inspection of the data showed that the dataset contains exploration data in areas where mineralisation that may constitute a Mineral Resource has not yet been identified and so all data east of 479,850 mX, north of 8,374,900 mY and deeper than 740 mZ were removed from the estimation dataset prior to any detailed analysis. Sporadic, narrow intersections above 0.5% occur outside of these limits but there is insufficient support from surrounding drillholes to define a Mineral Resource.

Five RC holes occur within the area. Information was not available on the sampling and assaying of the RC holes. Furthermore, these holes are shallow and would not contribute significantly to the estimate; therefore these holes were discarded from the estimation data. The discarded holes are as follows:

- KR2_R, KR3_R, KR4_R.
- KITRC_001, KITRC_002.

The final dataset that was used to inform the estimate consisted of the drillholes shown in Table 14.1

**Table 14.1 Drillholes used for Kitumba Mineral Resource Estimate**

| Prefix | Drillhole Number |
|---|---|
| KITDD_ | 01 to 40 |
| KR | 1_D |
| S1_ | 001-002 |
| S36__ | 01, 03-18, 20-36, 38 |
| ZMMUM | 0001, 0004, 0005 |

Eighty one drillholes with a combined length of 45,587 m were used for the estimate. Two of the drillholes only had lithology data (S1_001, S1_002), so were not used for grade estimation.

### 14.3.2 Validation of the database

The validation process consisted of:

- Examining the sample assay, collar survey, down-hole survey and geology data to ensure that the data were complete for all of the drillholes.

- Examining the de-surveyed data in three dimensions to check for spatial errors.

- Examination of the assay data in order to ascertain whether they were within expected ranges.

- Checks for 'FROM-TO' errors, to ensure that the sample data did not overlap one another or that there were no unexplained gaps between samples.

The validation exercise revealed the following:

- Down-hole survey data were not available for the following drillholes:

    - KITDD_001 (collar to end of hole 329.72 m). Collar survey (vertical) used for entire hole.

    - S36_033 (collar to end of hole 976.15 m). Collar survey (vertical) used for entire hole.

    - ZMMUM0004 (collar to end of hole 812 m). Collar survey (60° toward 354°) used for entire hole.

    - ZMMUM0005 (collar to end of hole 732 m). Collar survey (60° toward 354°) used for entire hole.

  The drillholes were included into the estimation database without down-hole survey data for the following reasons:

    - KITDD_001 intersected low grade mineralisation within the Mumbwa Fault Zone, east of the area that was reported as a Mineral Resource.

    - Although S36_033 was drilled in the high grade core of the Mineral Resource, it was later twinned by S36_038 that was collared 5 m away and also drilled vertically and was surveyed. Using the collared angle for the entire length of S36_033, the horizontal distance between S36_033 and S36_038 is approximately 10 m indicating that the deviation from vertical is approximately 5 m in this area, which is not considered material to this estimate.

    - Both ZMMUM0004 and ZMMUM005 are peripheral holes that intersected low grade mineralisation south of the area that was reported as a Mineral Resource.

- Examination of the drillhole data in three dimensions indicated no material spatial errors occur, the collars plotting close to the topographic surface.

- The SG data collected from two of the drillholes were found to be considerably higher than the surrounding holes. There was no geological reason for these anomalous data and thus the SG's from S36_007 and S36_009 were removed from the estimation database. There were also a number of SG data that fell outside of expected ranges for the Kitumba mineralisation and lithologies and all SG data less than 1.0 and greater than 5.4 were removed. This upper limit was based on a sample of 95% magnetite and 5% native copper. Samples with greater than 5% copper and SG greater than 5.4 were examined for the presence of significant native copper. If no native copper was logged the SG's of these samples would also be discarded. None were found.

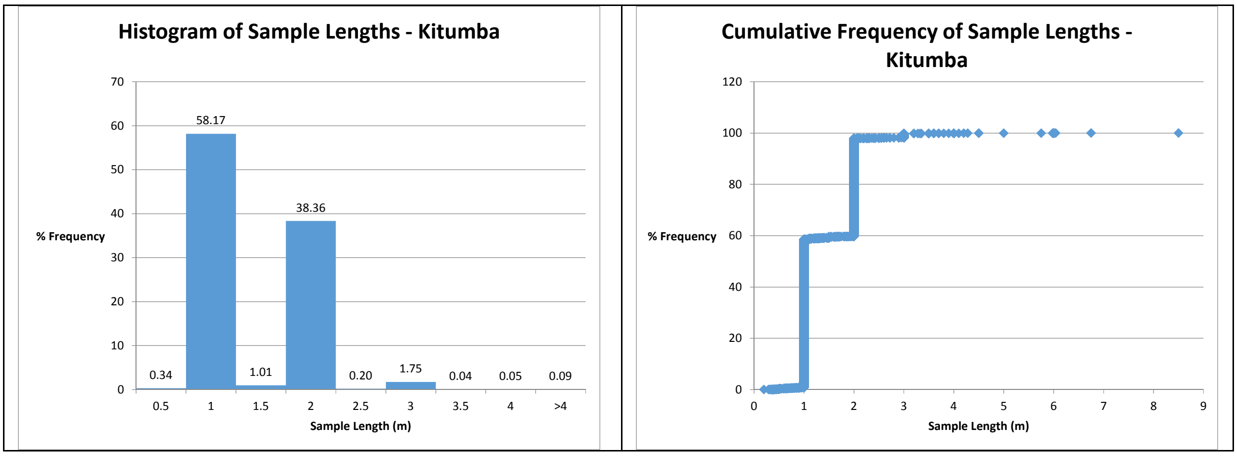
- During the exploratory statistical analysis, grade data were checked for those that were outside of expected ranges and none were found.

- There were no unexplained missing intervals or overlaps in the data.

### 14.3.3 Statistics of the Sample Data

A histogram and cumulative frequency plot of the sample lengths are presented in Figure 14.1. 98% of the samples are 2 m in length or less, which is the minimum sample length that should be considered for data compositing. The most frequent nominal sample lengths used were 1 m and 2 m.

**Figure 14.1 Histogram and Cumulative Frequency Plot of the Sample Length Data**




### 14.3.4 Statistics of the Assay Data

***Univariate Analysis***

A summary of the sample assay data statistics within the modelled area is shown in Table 14.2.

**Table 14.2 Summary of the Sample Data at Kitumba**

| Variable | Number of Assays | Mean Value | Minimum Value | Maximum Value |
|---|---|---|---|---|
| CUPC | 44866 | 0.57 | 0.00 | 42.69 |
| CUASPC | 27580 | 0.81 | 0.00 | 22.21 |
| COPPM | 31799 | 104 | 0 | 6707 |
| AUGT | 37543 | 0.04 | 0.00 | 4.57 |
| AGGT | 31757 | 0.84 | 0.01 | 165 |
| UPPM | 37715 | 36 | 0 | 3029 |
| FEPC | 37715 | 14.5 | 0.0 | 75.5 |
| MNPPM | 37357 | 5796 | 3 | 163110 |
| SPPM | 37070 | 8241 | 25 | 284700 |
| SG | 33193 | 2.69 | 1.05 | 5.19 |

Not all of the samples were assayed for the elements of interest, particularly during the earlier drilling campaigns. All of the samples that were assayed were assayed for CUPC. Less low grade copper samples were assayed for acid soluble copper, as evidenced by the higher mean value for CUASPC compared with CUPC.

The sample assay data were investigated lithologically in order to attempt to determine mineralisation relationships with rock type and texture. The mean of the sample data for each attribute of interest and each main lithology (LITH1 code) are presented in Table 14.3.

**Table 14.3 Sample Assay Data Mean Values**

| LITH1 | Number of Intervals with CUPC Assays | CUPC | CUASPC | COPPM | AUGT | AGGT | UPPM | FEPC | MNPPM | SPPM | SG |
|---|---|---|---|---|---|---|---|---|---|---|---|
| BREC | 8368 | 0.26 | 0.08 | 93 | 0.06 | 1.1 | 55 | 16.6 | 6812 | 5764 | 2.77 |
| FPSY | 19322 | 0.84 | 0.37 | 130 | 0.03 | 0.6 | 26 | 13.0 | 5717 | 12168 | 2.68 |
| IORE | 2330 | 0.18 | 0.09 | 107 | 0.12 | 0.8 | 99 | 32.4 | 8054 | 1569 | 3.01 |
| OVER | 92 | 0.09 | 0.01 | 170 | 0.03 | 2.6 | 42 | 21.3 | 7805 | 220 | 2.76 |
| QFPG | 10241 | 0.50 | 0.34 | 81 | 0.04 | 0.9 | 25 | 11.2 | 4470 | 5892 | 2.76 |
| SAPR | 308 | 0.10 | 0.01 | 104 | 0.04 | 1.7 | 38 | 17.9 | 6610 | 557 | 2.59 |
| SYEN | 2121 | 0.60 | 0.20 | 135 | 0.04 | 0.9 | 38 | 15.8 | 7783 | 10341 | 2.75 |

The lithologies containing the highest grade copper and cobalt mineralisation are those described as feldspar porphyry syenite (FPSY), syenite (SYEN) and quartz feldspar porphyry granite (QFPG). Samples described as breccia (BREC) are well mineralised with copper in places, but the mineralisation in this rock type is variable. Iron oxide replaced rock (IORE) is generally poorly mineralised with copper although sporadic high grades can occur.

The highest grade gold and uranium mineralisation occurs in IORE.

Silver grades are slightly enhanced in overburden (OVER) and saprolite (SAPR), although anomalous silver grades can occur in any of the lithologies at Kitumba. High silver grades are less spatially restricted than the copper, gold and uranium grades and tend to occur sporadically throughout the deposit.

High manganese grades occur together with high iron, and high sulphur grades occur in the lithologies containing the highest copper. Density does not vary much between lithologies and IORE is the densest lithology.

The grades of samples with brecciated textures were examined. For most metals there were no consistent differences between the grades in brecciated and un-brecciated textures. However, both gold and uranium grades were consistently higher in the samples logged as brecciated.

***Bivariate Analysis***

The sample assay data of the individual elements and SG were compared with each other in order to understand the relationships between the various attributes that were estimated. Table 14.4 presents a summary of the relationships that were found.

**Table 14.4 Summary of the Bivariate Analysis**

| | CUPC | CUASPC | COPPM | AUGT | AGGT | UPPM | FEPC | SG |
|---|---|---|---|---|---|---|---|---|
| CUPC | - | Capped | Slight Inverse | Strong Inverse | None | Strong Inverse | Slight Inverse | None |
| CUASPC | | - | Moderately Inverse | Strongly Inverse | None | Strongly Inverse | Slight Inverse | None |
| COPPM | | | - | Inverse | Slight Inverse | Slight Inverse | None | None |
| AUGT | | | | - | Slight Inverse | None | None | None |
| AGGT | | | | | - | Mod Inverse | Slight Inverse | None |
| UPPM | | | | | | - | Slight positive linear | None |
| FEPC | | | | | | | - | Positive linear |
| SG | | | | | | | | - |

None of the attributes examined exhibited strong linear relationships with one another, indicating that the grade of the deposit is likely to be zoned with enhanced grades of individual elements occurring in different areas. The bivariate analysis was refined by discriminating by the main mineralised rock types, however the same relationships were observed as for the total data.

The relationship between CUPC and CUASPC is not linear. A linear relationship should not be expected in this environment given the array of oxidation states present in the deposit and the transition from leached through to hypogene. In a few instances, the CUASPC was higher than the CUPC, although one extreme sample returned a grade of 28.17% CUASPC and 10.14% CUPC.

Most of the differences were within the limits of assay error and where acid soluble copper was higher than total copper, the values were adjusted to be equal to the CUPC value.

The relationship between copper, gold and uranium in the well mineralised lithologies (FPSY, SYEN, BREC), indicates that there are two mineralisation populations within these rock types; that containing higher grade copper or that containing higher grade gold or uranium. In other words, if high grade copper is intersected one should expect low grade gold and uranium. It is not a negative linear correlation but rather an either/or relationship (an inverse relationship).

For IORE and FPGR there are no discernible relationships between the individual elements of interest, i.e. the relationship is random. Density tends to increase with increasing iron content in IORE, although the scatter is high and correlation therefore poor.

***Exploratory Spatial Analysis***

Examination of the metal grades spatially (in X, Y and Z directions), by plotting the coordinates against the metal grades on a scatterplot, shows that:

The high copper grades (>5%) are restricted to a narrow area both in the X and Y directions and with depth (Figure 14.2). In the X direction, the higher grades tend to occur over an interval of approximately 200 m and over a broader zone in the Y direction. So far in the drilling program there are no values of > 2% copper encountered above the 840 m elevation or below the 1,320 m elevation. The high grade mineralisation forms an irregular elongate elliptical shape that is orientated with its longest axis approximately northwards and its shortest axis from east to west.

The high grade copper zone occurs in excess of 100 m below surface, indicating the presence of a deeply leached zone.

The particularly high grade acid soluble copper tends to occur over a more restricted area than the total copper.

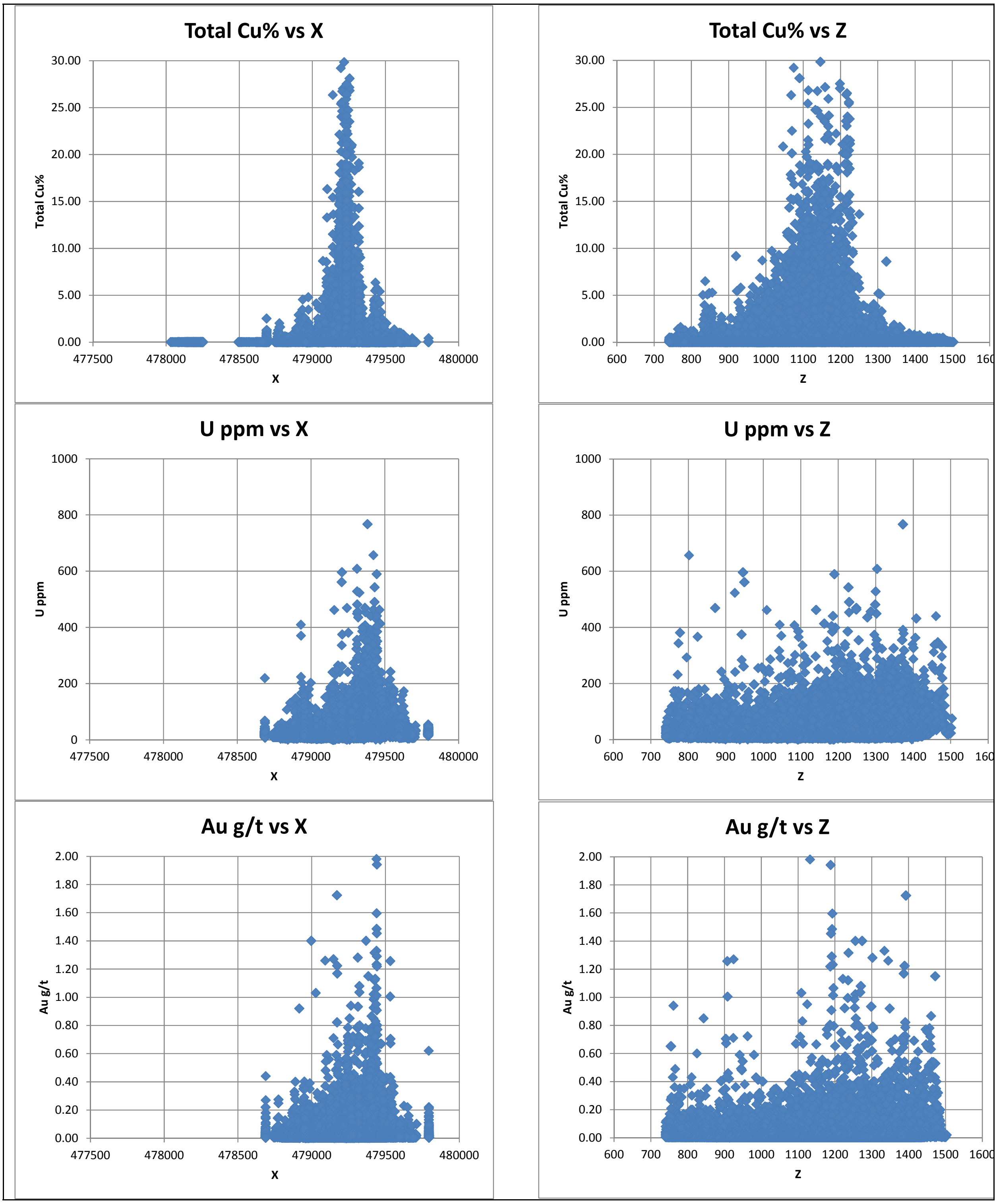
The higher uranium grades tend to occur in the same area as the high iron grades.

The higher uranium values tend to be offset from the higher copper values by approximately 100 m to the east.

Higher grade gold and silver values are more closely associated with higher uranium values than copper; however they are more dispersed, particularly silver.

When viewing the rock type and grade data in three dimensions, it is apparent that the higher grade zone does indeed have an approximate elongate elliptical shape but is quite irregular in some areas with protrusions to the west and a fairly sharp cut-off to the east. SYEN and FPSY occur in a zone sandwiched between IORE grading into breccias in the east and QFPG in the west. Copper grades tend to decrease sharply in the IORE eastwards where copper grades in the IORE samples rarely reach more than 1.5%. Westwards into the QFPG dominant area, grades tend to be lower than in the syenite dominant area, although sporadic high grades also occur.

**Figure 14.2 Scattergrams of Various Elements Against Coordinates Illustrating Spatial Grade Separation**



#### 14.3.5 Summary of the Exploratory Analysis of the Raw Dataset

Most of the samples are less than 2 m long, which should be the minimum length considered for sample compositing.

The host rocks to most of the copper mineralisation are described as syenite and porphyry syenite as well as to a lesser extent the quartz feldspar porphyry granite to the west. Sporadic elevated copper grades occur in the breccia but rarely above 1.5% Cu.

The host rocks tend to exhibit an either/or relationship in that if they contain high grade copper they are unlikely to contain high grade gold, silver or uranium and vice-versa.

The IORE zone to the east tends not to contain high grade copper mineralisation.

The high grade copper zone forms an irregular elongate elliptical shape with its long axis approximately north south and its shortest axis east to west. The very high grades so far intersected by the drilling occur from the 840 m elevation to the 1,320 m elevation, which is in excess of 100 m below surface, indicating that a deep leached zone occurs.

There is no linear relationship between total copper and acid soluble copper.

Copper and cobalt are spatially related although there is no clear linear relationship between them.

The higher uranium, gold and silver values are also spatially related but do not exhibit a clear linear relationship. They tend to occur over a broader area than the high grade copper zone and the highest values are offset approximately 100 m to the east of the high grade copper zone. The higher Au and U grades are not concentrated in a depth range and occur from surface.

The estimation parameters and methodology need to take careful consideration of the restricted nature of the high grade copper zone, the broad nature of the precious metal and uranium zone and the existence of the leached zone. The leached zone does not affect Au, Ag and U grade.

The different attributes do not have a strong relationship with one another and can be estimated independently.

## 14.4 Geological Modelling

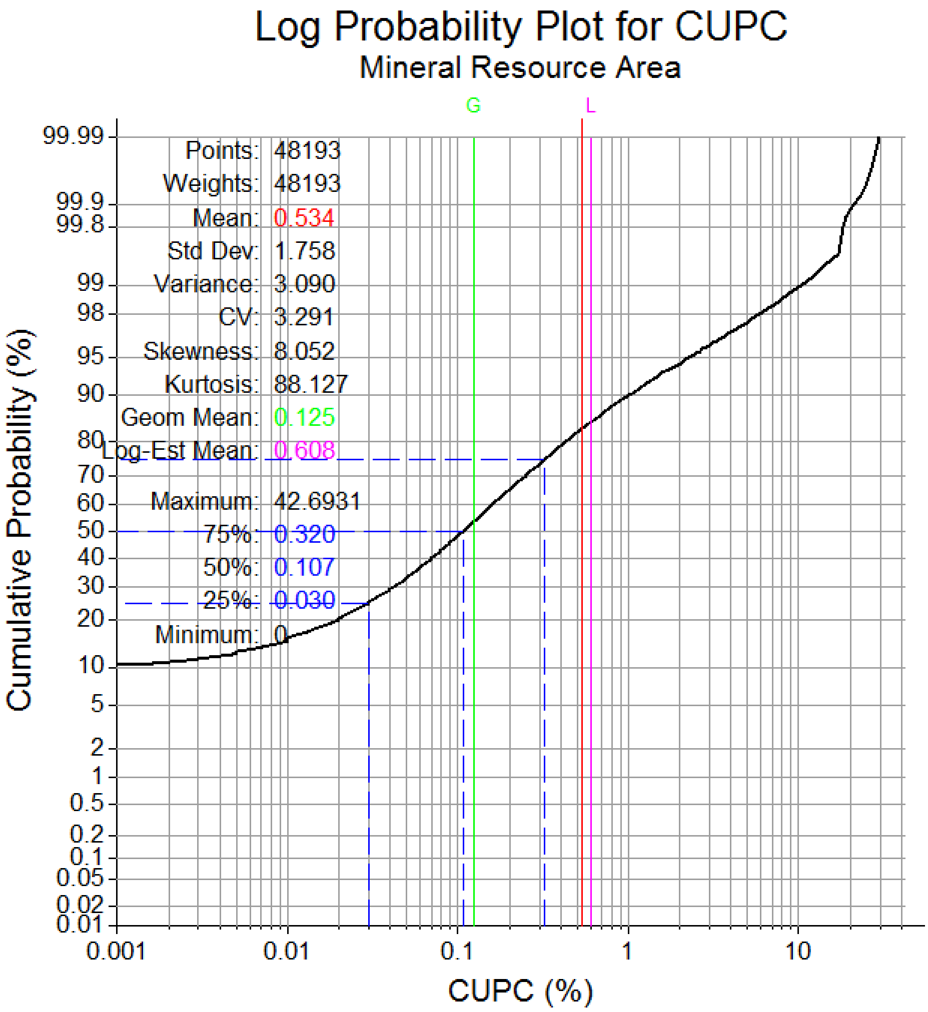
The topographic surface provided by BRL was based on a 25 m by 25 m point grid. These were converted into a digital terrain model (DTM) that was used to constrain the surface of the block model.

#### 14.4.1 Domain Wireframes

Mineralogical zones (leached, supergene, transitional and hypogene) were identified by recording the presence of various species of copper minerals in the drillhole cores. The supergene domain was logged based on the presence of malachite and chalcocite, and the hypogene domain was logged based on the presence of chalcopyrite. The leached style of mineralisation was logged from near surface to in excess of 100 m below surface, however a zone of poor grade Cu mineralisation was identified that extends below the visually identified leached zone to between 150 m and 200 m below surface. The distribution of the transitional style of mineralisation is irregular and, for estimation purposes, was mostly incorporated into the supergene zone. For this reason and that the categorisation of the domains was based on mineral type prevalence, the supergene and hypogene domains will both contain sulphide and oxide mineralisation; the hypogene domain being sulphide dominant and the supergene domain being oxide dominant. Both the leached-supergene and supergene-hypogene modelled surfaces are very irregular, reflecting the complex mineralisation environment at Kitumba. Peaks and troughs in these surfaces are likely to have a strong structural control thus explaining the vertical component.

Thresholds for constraining the grade estimate were considered by examination of the log probability plot of the raw data. No obvious grade thresholds were observed (Figure 14.3) and the breaks in the log probability plots are subtle and possibly occur in the region of 0.5% CUPC and 1.5% CUPC. Examination of the 1.5% threshold found that this threshold was not sharp and would add little value to the quality of the estimate. However, a high grade core to the deposit was observed, which was visually best defined by a 4% CUPC threshold. A slightly more relaxed grade threshold of 0.30% CUPC than indicated by the log probability plot, was chosen in order to constrain the Cu grade estimate, as the 0.5% threshold was considered too close to a potential open-pit cut-off grade to be used. The aforementioned grade domains were used to control CUPC, CUASPC, SPPM and density estimation.

**Figure 14.3 Log Probability Plot of the Total Cu Data**

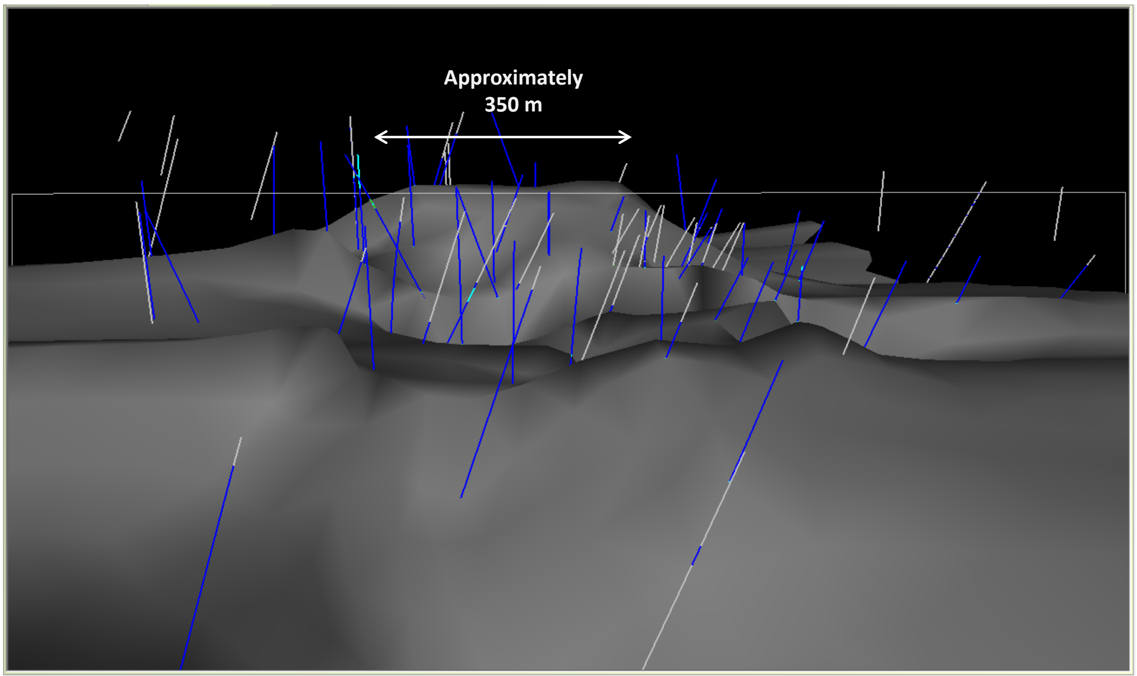



***Leached / Low Copper Domain***

This domain was identified by visual examination of the drillhole data. A point in each drillhole was identified at the depth when significant Cu mineralisation was first observed down the hole, which generally corresponds to a grade in the region of 0.30% CUPC. A DTM was constructed from the points (Figure 14.4). The surface is undulating with a number of troughs and depressions.

**Figure 14.4 Isometric View of the Modelled 0.30% Cu Upper Surface (base of low Cu Zone)**

**View is looking approximately towards the South**



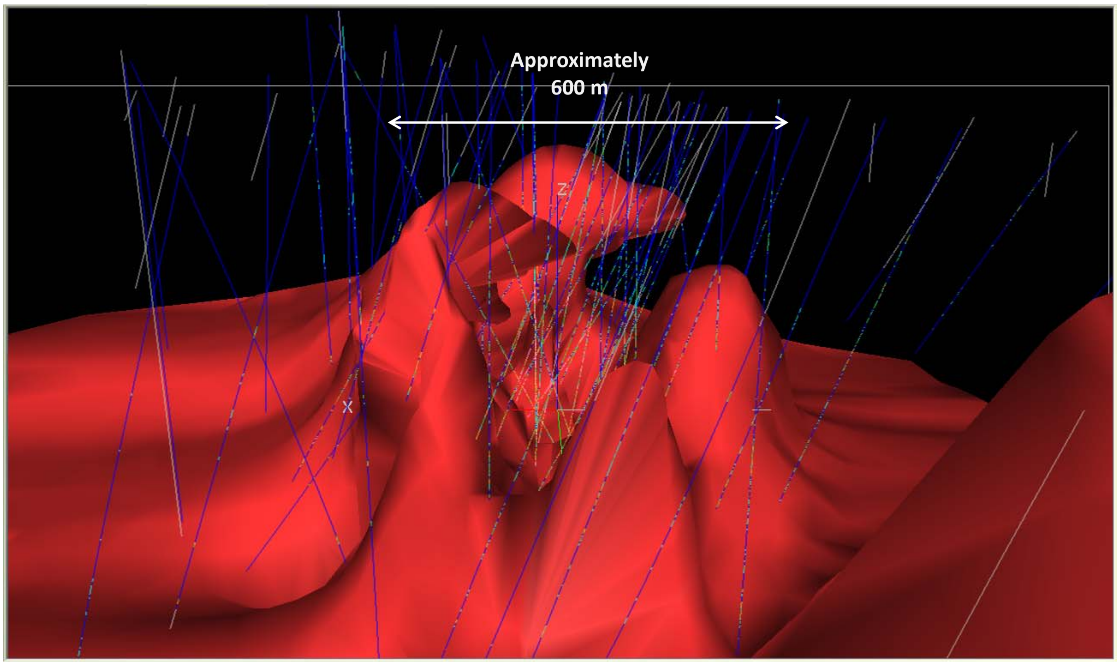

***Not to scale: The approximate maximum dimension of the "flat top hill" like structure area from left to right is 350 m in the view shown***

### *Hypogene – Supergene Interface*

By using the mineralisation type codes, as assigned to the core samples by the site geologists, a surface representing the interface between supergene and hypogene mineralisation was defined. A point in each drillhole was identified at the depth when the hypogene mineralisation became dominant over supergene mineralisation down the hole. Where a transitional mineralisation type was identified, this was normally taken as supergene. A DTM was constructed from the points (Figure 14.5). The resulting DTM is irregular with deep depressions in the core of the deposit coinciding with the highest grade supergene mineralisation.

**Figure 14.5 Isometric View of the Modelled Supergene-Hypogene Interface Surface**

**View is looking approximately towards the South**




***Not to scale: The approximate maximum dimension of the hypogene peak area from left to right is 600 m in the view shown***

### *0.30% Cu Grade Shell*

An Indicator Kriging approach was used to model a 0.30% CUPC grade shell. The procedure used is as follows:

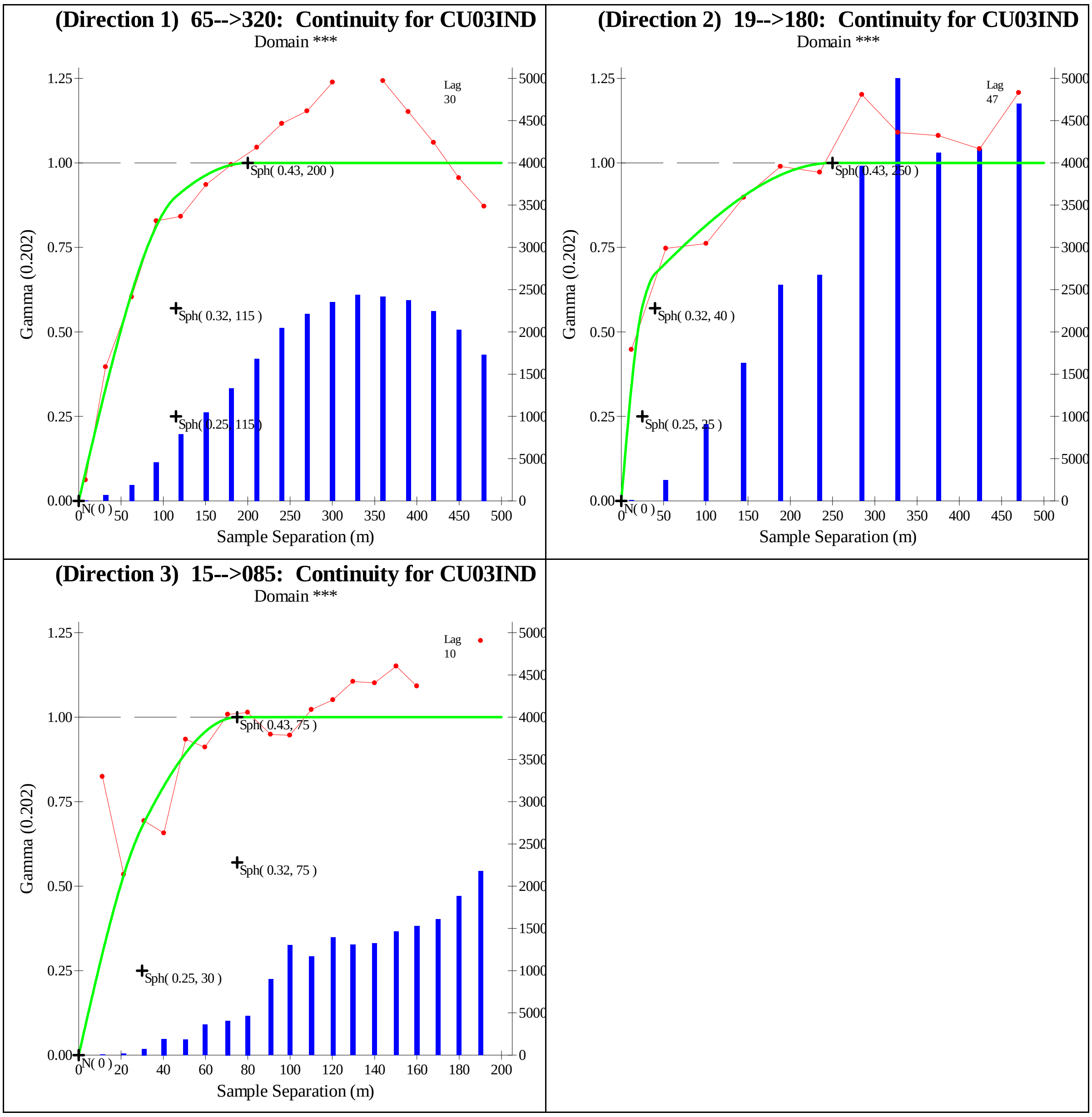
- Composite the drillhole data to 5 m lengths.
- Assign indicators of 1 (greater than 0.30% CUPC) or 0 (less than 0.30% CUPC) to the composites.
- Calculate and model a variogram of the indicators.
- Create a block model 10 mX, 20 mY and 20 mZ.
- Estimate the indicators into the block model.
- Create wireframes containing block model cells with an indicator value of greater than 0.5, i.e. a probability of more than 50% that the block will have a grade of greater than 0.30% CUPC. The wireframes were constructed using the drillhole data to ensure that the resulting model correctly honoured the input data. When constructing the grade shell wireframes lower probability indicator values were considered in order to create smoother surfaces where possible.

The search parameters used for the Indicator Kriging are shown in Table 14.5 and the indicator variograms in Figure 14.6. The search distance was restricted to the variogram range in each direction. The search ellipse was orientated steeply dipping to the east with a shallow plunge to the north. The longest axis was orientated plunging towards 5° west of north and the shortest axis just off east to west.

**Table 14.5 Search Parameters for Indicator Kriging of the 0.30% CUPC Threshold**

| SDIST1 | SDIST2 | SDIST3 | SANGLE1 | SANGLE2 | SANGLE3 | MINNUM | MAXNUM |
|---|---|---|---|---|---|---|---|
| 200 | 250 | 75 | 85 | 75 | 110 | 4 | 12 |
| **SAXIS1** | **SAXIS2** | **SAXIS3** | | | | | |
| 3 | 1 | 3 | | | | | |

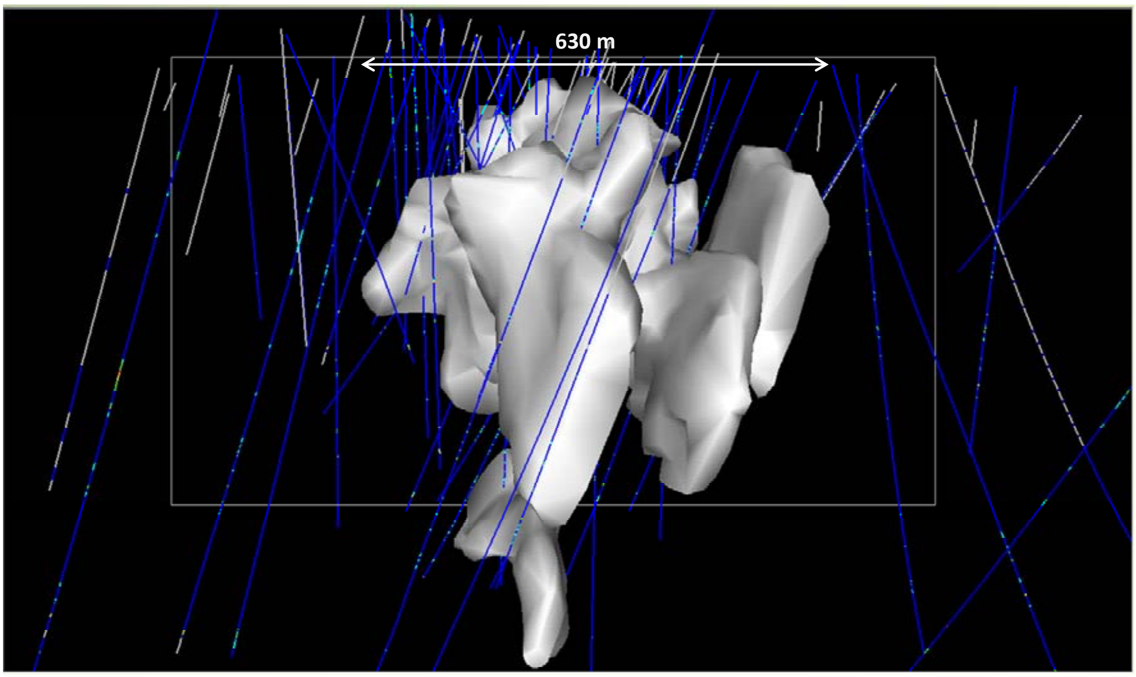
**Figure 14.6 0.30% CUPC Indicator Variograms**




The resulting 0.30% CUPC grade shell was quite irregular with a strong north-south orientation, a steep dip to the east and a number of protrusions to the west (Figure 14.7).

**Figure 14.7 Isometric View of the Modelled 0.30% Cu Grade Shell**

**View is looking approximately towards the South**



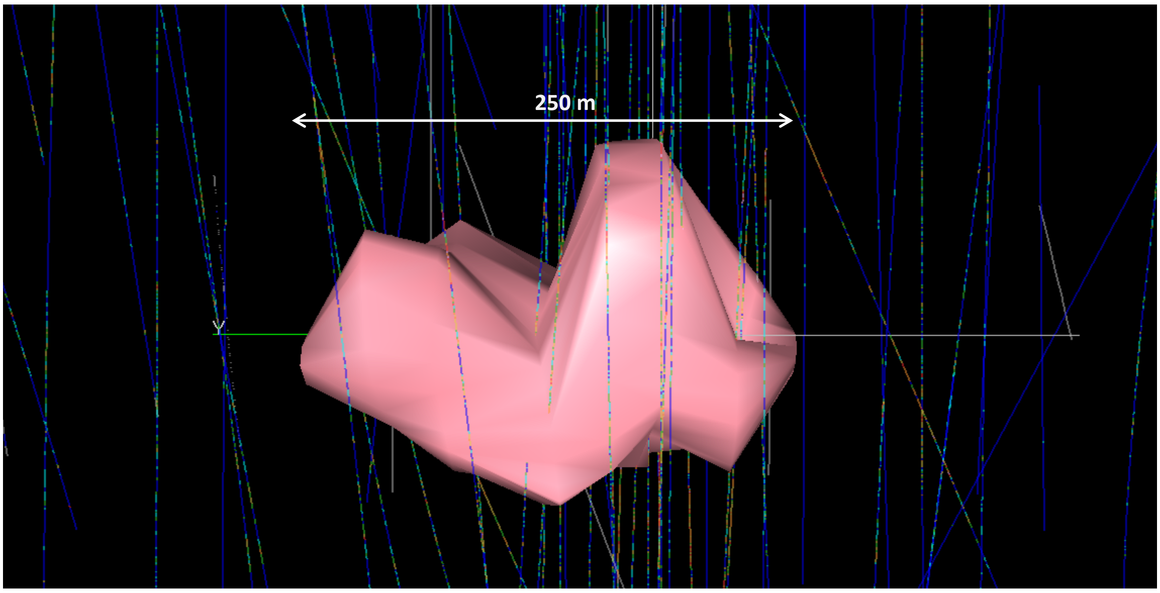

***Not to scale: The approximate maximum dimension of the 3D solid from left to right is 630 m in the view shown***

### *4.0% Cu Grade Shell*

This domain was identified by visual examination of the drillhole data. Perimeters were digitised around the drillhole samples forming a cohesive zone of greater than 4% Cu. The perimeters were linked to form a solid wireframe. The 4% Cu grade shell extends for a maximum of approximately 250 m along strike, 150 m in depth and is on average approximately 100 m wide. The 4% Cu grade shell lies largely within the deep trough in the Supergene-Hypogene surface and is mostly contained within the Supergene Zone.

**Figure 14.8 Isometric View of the Modelled 4% Cu Grade Shell**

**View is looking approximately towards the East**




***Not to scale: The approximate maximum dimension of the 3D solid from left to right is 250 m in the view shown***

### 14.4.2 Summary of Estimation Domains

Three grade domains were identified:

- Domain 0 = outside 0.30% Cu grade shell (waste domain)
- Domain 1 = inside 0.30% Cu grade shell, outside 4% shell
- Domain 2= inside 4% Cu grade shell (occurs within 0.30% grade shell).

Three mineralisation domains were also identified:

- LCU = near surface "leached" domain with less than 0.3% Cu
- SUP = supergene domain, based on logging (presence of malachite and chalcocite)
- HYP = hypogene domain, based on logging (presence of chalcopyrite).

Examination of the data showed that there is spatial separation between high grade copper mineralisation and many of the other elements of interest. Once the mineralisation wireframes were constructed, it was also observed that grades other than CUPC, CUASPC and SPPM were not controlled by the mineralisation state. However, SG values were different between the mineralisation type domains. Therefore the supergene, hypogene and leached/low copper zones were only used to separate estimation domains for CUPC, CUASPC, SPPM and density.

In total six estimation domains were created:

- LCU.
- Supergene outside 0.30% Cu grade shell (Domain 0 + SUP).
- Supergene inside 0.30% Cu grade shell but outside 4% Cu grade shell (Domain 1 + SUP).
- Hypogene outside 0.30% Cu grade shell (Domain 0 + HYP).
- Hypogene inside 0.30% Cu grade shell but outside 4% Cu grade shell (Domain 1 + HYP).
- Inside 4% Cu grade shell mostly in supergene (Domain 2). Estimated as a single domain due to few hypogene data but coded as SUP or HYP.

## 14.5 Statistical Analysis of the Composite Data

Data were coded for each mineralisation type using the mineralisation type wireframes and the leached/low copper zone.

The sample data were composited into two metre intervals and where applicable by mineralisation type and by grade domain. Compositing was carried out so that none of the sample data was discarded, which gives composite lengths close to, but not exactly the chosen length.

### 14.5.1 Statistical Analysis

The composite data were de-clustered by the same dimensions as the estimation block model (20 mX, 40 mY, 10 mZ). Data were interrogated for each estimation domain. A summary of the de-clustered statistics is shown in Table 14.6 and the histograms and log probability plots are shown in Appendix 3. For CUPC the coefficients of variation (CV's) are between 0.93 and 1.46, whereas for CUASPC the CV's are higher (between 1.48 and 3.43). In the Supergene and leached zone the CV's for SPPM are high (between 1.86 and 2.56) but for the hypogene zone the CV's for sulphur are moderate, between 0.77 and 1.10. The high sulphur grade CV in the supergene zone reflects many low sulphur samples combined with small hypogene high sulphur zones of mineralisation within the supergene and transitional mineralisation domains. The CV's for gold and silver are high (2.19 and 2.29 respectively) whereas manganese, iron and uranium exhibit moderate CV's (between 0.60 and 1.42). SG has low CV's. Most histograms are strongly positively skewed with the exception of SG that has a near bell shaped histogram and gold, silver and uranium that are strongly positively skewed.

**Table 14.6 Summary Statistics (de-clustered) of the Estimation Data**

| | Number of Composites | Min | Max | Mean | Geometric Mean | CV | Skewness |
|---|---|---|---|---|---|---|---|
| **Leached/Low Cu Domain** | | | | | | | |
| CUPC | 2882 | 0.00 | 0.61 | 0.07 | 0.05 | 0.96 | 2.1 |
| CUASPC | 1721 | 0.00 | 0.19 | 0.02 | 0.01 | 1.48 | 3.7 |
| SPPM | 2250 | 25 | 35,150 | 1,511 | 322 | 2.56 | 4.5 |
| SG | 2507 | 1.27 | 4.57 | 2.63 | 2.61 | 0.12 | 0.8 |
| **Supergene >0.30 <4.0% Cu Domain** | | | | | | | |
| CUPC | 3028 | 0.01 | 18.24 | 0.70 | 0.43 | 1.37 | 5.4 |
| CUASPC | 2596 | 0.00 | 10.28 | 0.25 | 0.09 | 2.36 | 7.5 |
| SPPM | 2196 | 25 | 119,963 | 6,520 | 1300 | 1.86 | 3.2 |
| SG | 2656 | 1.76 | 4.71 | 2.59 | 2.57 | 0.10 | 1.2 |
| **Supergene < 0.30% Cu Domain** | | | | | | | |
| CUPC | 6358 | 0.00 | 3.93 | 0.13 | 0.08 | 1.23 | 5.7 |
| CUASPC | 3348 | 0.00 | 1.28 | 0.04 | 0.02 | 1.43 | 7.6 |
| SPPM | 5621 | 25 | 100,000 | 2714 | 457 | 2.33 | 5.0 |
| SG | 4567 | 1.11 | 4.57 | 2.69 | 2.68 | 0.11 | 1.2 |
| **Hypogene >0.30 <4.0% Cu Domain** | | | | | | | |
| CUPC | 2485 | 0.00 | 14.37 | 0.80 | 0.44 | 1.28 | 3.4 |
| CUASPC | 2246 | 0.00 | 12.83 | 0.10 | 0.04 | 3.43 | 23.3 |
| SPPM | 1547 | 51 | 129,526 | 23,081 | 15,769 | 0.77 | 1.8 |
| SG | 2089 | 1.84 | 4.03 | 2.83 | 2.82 | 0.08 | -0.4 |
| **Hypogene <0.30% Cu Domain** | | | | | | | |
| CUPC | 4174 | 0.00 | 3.43 | 0.10 | 0.05 | 1.46 | 5.8 |
| CUASPC | 3067 | 0.00 | 0.63 | 0.02 | 0.01 | 1.66 | 5.7 |
| SPPM | 3438 | 25 | 98,600 | 9,927 | 4,910 | 1.10 | 2.8 |
| SG | 3523 | 1.77 | 4.14 | 2.82 | 2.81 | 0.08 | 0.8 |
| **>4% Cu Domain** | | | | | | | |
| CUPC | 832 | 0.09 | 33.0 | 5.34 | 3.31 | 0.93 | 1.7 |
| CUASPC | 813 | 0.02 | 20.1 | 2.54 | 1.07 | 1.24 | 2.2 |
| SPPM | 483 | 70 | 129,505 | 22,284 | 9,514 | 1.09 | 1.5 |
| SG | 693 | 1.67 | 3.68 | 2.52 | 2.51 | 0.08 | 0.5 |
| **All Domains** | | | | | | | |
| COPPM | 13514 | 1 | 2978 | 96 | 65 | 1.16 | 8.0 |
| AUGT | 16739 | 0.00 | 4.57 | 0.04 | 0.02 | 2.19 | 16.0 |
| AGGT | 13485 | 0.01 | 108.2 | 0.9 | 0.4 | 2.29 | 16.0 |
| UPPM | 15941 | 1 | 2,964 | 36 | 25 | 1.42 | 16.8 |
| FEPC | 15941 | 0.6 | 71.2 | 14.1 | 11.9 | 0.60 | 1.7 |
| MNPPM | 15641 | 6 | 119,632 | 6,102 | 2,503 | 1.15 | 2.8 |

### 14.5.2 Cutting and Capping

The log probability plots and histograms were examined for outlier values that have a low probability of re-occurrence. These values were capped to a threshold as shown in Table 14.7.

The capping applied was not severe and no significant change in the mean value occurred; however a slight reduction in the CV was achieved.

**Table 14.7 Impact of Capping the Estimation Data**

| | Before Capping | | | After Capping | | | |
|---|---|---|---|---|---|---|---|
| | Number of Composites | Mean | CV | Cap Value | Number of Composites Capped | Mean | CV |
| **Low Cu/Leached Domain** | | | | | | | |
| CUPC | 2882 | 0.07 | 0.96 | - | - | - | - |
| CUASPC | 1721 | 0.02 | 1.48 | - | - | - | - |
| SPPM | 2250 | 1,511 | 2.56 | - | - | - | - |
| SG | 2507 | 2.63 | 0.12 | - | - | - | - |
| **Supergene >0.30%, <4.0% Cu Domain** | | | | | | | |
| CUPC | 3028 | 0.70 | 1.37 | 10.0 | 3 | 0.70 | 1.33 |
| CUASPC | 2596 | 0.25 | 2.36 | 10.0 | 1 | 0.25 | 2.35 |
| SPPM | 2196 | 6,520 | 1.86 | - | - | - | - |
| SG | 2656 | 2.59 | 0.10 | - | - | - | - |
| **Supergene <0.30% Cu Domain** | | | | | | | |
| CUPC | 6358 | 0.13 | 1.23 | 2.5 | 1 | 0.13 | 1.20 |
| CUASPC | 3348 | 0.04 | 1.43 | 0.5 | 2 | 0.04 | 1.30 |
| SPPM | 5621 | 2,714 | 2.33 | 55,000 | 4 | 2,686 | 2.23 |
| SG | 4567 | 2.69 | 0.11 | - | - | - | - |
| **Hypogene >0.30% Cu, <4.0% Cu Domain** | | | | | | | |
| CUPC | 2485 | 0.80 | 1.28 | 7.5 | 2 | 0.80 | 1.25 |
| CUASPC | 2246 | 0.10 | 3.43 | 2.7 | 6 | 0.09 | 2.25 |
| SPPM | 1547 | 23,081 | 0.77 | - | - | - | - |
| SG | 2089 | 2.83 | 0.08 | - | - | - | - |
| **Hypogene <0.30% Cu** | | | | | | | |
| CUPC | 4174 | 0.10 | 1.46 | 1.5 | 3 | 0.10 | 1.38 |
| CUASPC | 3067 | 0.02 | 1.66 | 0.35 | 1 | 0.02 | 1.59 |
| SPPM | 3438 | 9,927 | 1.10 | - | - | - | - |
| SG | 3523 | 2.82 | 0.08 | - | - | - | - |
| **>4.0% Cu** | | | | | | | |
| CUPC | 832 | 5.34 | 0.93 | - | - | - | - |
| CUASPC | 813 | 2.54 | 1.24 | - | - | - | - |
| SPPM | 483 | 22,284 | 1.09 | - | - | - | - |
| SG | 693 | 2.52 | 0.08 | - | - | - | - |
| **All Domains** | | | | | | | |
| COPPM | 13514 | 96 | 1.16 | 1,200 | 15 | 95 | 1.04 |
| AUGT | 16739 | 0.04 | 2.19 | 1.1 | 16 | 0.04 | 1.84 |
| AGGT | 13485 | 0.9 | 2.29 | 22 | 10 | 0.9 | 1.90 |
| UPPM | 15941 | 36 | 1.42 | 500 | 10 | 36 | 1.18 |
| FEPC | 15941 | 14.1 | 0.60 | - | - | - | - |
| MNPPM | 15641 | 6,102 | 1.15 | 58,000 | 6 | 6,092 | 1.14 |

## 14.6 Geostatistical Analysis

### 14.6.1 Variography

Variography was conducted using Snowden Supervisor software.

Variograms were calculated on the 2 m composite data for all attributes of interest. Variograms for CUPC, CUASPC and SPPM were calculated for the respective domains within the >0.3% Cu grade shell and the near surface low copper domain. Variograms for the other elements were calculated for the data within the total estimation area.

Normalised variograms were calculated so that the sum of the variance is equal to one. Several of the variograms were calculated using data that was transformed to normal scores. The modelled variances were back transformed prior to kriging.

Variograms were calculated in three directions on the uncapped data. Some of the variogram structures benefitted from data cuts by removal of detection limit values and some of the more extreme values. The cuts used for variography are shown in Table 14.8.

**Table 14.8 Cuts Applied for Variography**

| | Cut Value | | Number of composites cut | |
|---|---|---|---|---|
| | **Bottom Cut** | **Top Cut** | **Bottom Cut** | **Top Cut** |
| **Leached/Low Cu Domain** | | | | |
| CUPC | - | - | - | - |
| CUASPC | - | - | - | - |
| SPPM | - | 10000 | - | 85 |
| **Supergene** | | | | |
| CUPC | 0.01 | 15.5 | 3 | 13 |
| CUASPC | 0.01 | 5 | 75 | 70 |
| SPPM | - | - | - | - |
| **Hypogene** | | | | |
| CUPC | 0.01 | - | 7 | - |
| CUASPC | 0.01 | 1.8 | 280 | 8 |
| SPPM | - | - | - | - |
| **All Domains** | | | | |
| COPPM | - | 900 | - | 34 |
| AUGT | - | 0.65 | - | 47 |
| AGGT | 0.01 | 11 | 248 | 67 |
| UPPM | - | 350 | - | 39 |
| FEPC | - | - | - | - |
| MNPPM | - | 49000 | - | 52 |
| SG | - | - | - | - |

Variograms were modelled using one or more spherical structures. The nugget effect was estimated by extrapolation of the first two experimental variogram points, calculated at the same lag as the composite length, to the Y axis. In general, the variograms are robust with the models being informed by several experimental points in all three directions.

As far as considered reasonable, the variogram directions were aligned in the same directions for each element. However, analysis of the Kitumba data revealed that many of the attributes are not statistically or spatially related. Given the complex mineralisation history at Kitumba, it should be expected that the variograms will not be similar to each other for the respective metals and therefore several different directions of continuity were modelled, each metal behaving independently.

The variogram model parameters are shown in Table 14.9 and the variograms are presented in Appendix 3. The variogram sills modelled on normal scores experimental data have been back transformed as shown in Table 14.9.

**Table 14.9 Variogram Parameters - Kitumba**

| Attribute | Transform | Rotation Angle | | | Rotation Axis | | | Nugget Effect (C0) | Range of First Structure (R1) | | | Sill 1 (C1) | Range of Second Structure (R2) | | | Sill 2 (C2) | Range of Third Structure (R3) | | | Sill 3 (C3) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | 1 | 2 | 3 | 1 | 2 | 3 | | 1 | 2 | 3 | | 1 | 2 | 3 | | 1 | 2 | 3 | |
| **Low Cu/Leached Domain** | | | | | | | | | | | | | | | | | | | | |
| CUPC | Normal Scores | 90 | 75 | 155 | 3 | 1 | 3 | 0.11 | 220 | 10 | 40 | 0.21 | 220 | 110 | 40 | 0.24 | 220 | 110 | 145 | 0.44 |
| CUASPC | Normal Scores | 90 | 75 | 155 | 3 | 1 | 3 | 0.11 | 220 | 10 | 40 | 0.21 | 220 | 110 | 40 | 0.24 | 220 | 110 | 145 | 0.44 |
| SPPM | Normal Scores | 90 | 90 | 90 | 3 | 1 | 3 | 0.12 | 75 | 120 | 100 | 0.88 | - | - | - | - | - | - | - | - |
| **Supergene Domain** | | | | | | | | | | | | | | | | | | | | |
| CUPC | Normal Scores | 90 | 75 | 90 | 3 | 1 | 3 | 0.14 | 10 | 20 | 45 | 0.32 | 130 | 65 | 60 | 0.13 | 130 | 360 | 60 | 0.41 |
| CUASPC | Normal Scores | 90 | 75 | 90 | 3 | 1 | 3 | 0.10 | 10 | 20 | 25 | 0.29 | 95 | 50 | 60 | 0.18 | 95 | 360 | 60 | 0.43 |
| SPPM | Normal Scores | 90 | 75 | 90 | 3 | 1 | 3 | 0.07 | 20 | 35 | 25 | 022 | 320 | 35 | 125 | 0.18 | 320 | 1800 | 160 | 0.53 |
| **Hypogene Domain** | | | | | | | | | | | | | | | | | | | | |
| CUPC | Normal Scores | 110 | 65 | 100 | 3 | 1 | 3 | 0.18 | 125 | 160 | 22 | 0.18 | 125 | 160 | 60 | 0.64 | - | - | - | - |
| CUASPC | Normal Scores | 110 | 65 | 100 | 3 | 1 | 3 | 0.31 | 120 | 85 | 12 | 0.27 | 120 | 170 | 95 | 0.42 | - | - | - | - |
| SPPM | Normal Scores | 90 | 90 | 90 | 3 | 1 | 3 | 0.07 | 16 | 300 | 12 | 0.19 | 65 | 300 | 130 | 0.23 | 600 | 300 | 130 | 0.51 |
| **All Domains** | | | | | | | | | | | | | | | | | | | | |
| UPPM | Normal Scores | 90 | 70 | 90 | 3 | 1 | 3 | 0.08 | 30 | 25 | 50 | 0.37 | 800 | 60 | 170 | 0.14 | 800 | 1600 | 170 | 0.41 |
| AUGT | Normal Scores | 90 | 70 | 90 | 3 | 1 | 3 | 0.05 | 15 | 15 | 20 | 0.56 | 130 | 50 | 28 | 0.23 | 750 | 440 | 28 | 0.16 |
| AGGT | Normal Scores | 90 | 70 | 90 | 3 | 1 | 3 | 0.19 | 50 | 150 | 80 | 0.23 | 550 | 150 | 80 | 0.10 | 550 | 650 | 500 | 0.48 |
| FEPC | None | 90 | 70 | 90 | 3 | 1 | 3 | 0.07 | 25 | 100 | 60 | 0.31 | 130 | 150 | 280 | 0.12 | 520 | 650 | 280 | 0.50 |
| COPPM | None | 90 | 70 | 90 | 3 | 1 | 3 | 0.12 | 10 | 130 | 110 | 0.38 | 40 | 220 | 110 | 0.17 | 160 | 220 | 110 | 0.33 |
| MNPP | None | 90 | 70 | 90 | 3 | 1 | 3 | 0.14 | 9 | 300 | 100 | 0.17 | 260 | 300 | 200 | 0.69 | - | - | - | - |
| SG | None | 90 | 70 | 90 | 3 | 1 | 3 | 0.32 | 20 | 110 | 35 | 0.17 | 305 | 260 | 190 | 0.51 | - | - | - | - |

## 14.7 Kriging Neighbourhood Analysis

A Kriging Neighbourhood Analysis (KNA) was performed on the CUPC data in order to find a reasonable block size, number of composites for estimation and discretisation level. A single block model cell was created at a number of positions within the mineralised zone. The cell was expanded in various amounts in a number of directions until the highest Kriging Efficiency (KE%) and highest Slope of Regression (PSlope) were obtained. The most suitable block was chosen based on the block size with the highest KE% and PSlope, while considering likely mine design factors. Once the most suitable block size was chosen, the number of composites was increased while measuring KE%, PSlope, CUPC grade and the number of negative weights. These were plotted graphically and the minimum and maximum number of composites required for an estimate was obtained, while ensuring that an estimate was not attained with significant amounts of negative weights (normally more than two in a well-informed area). The discretisation level was varied using the optimal block size and numbers of composites until enough discretisation points were used so that KE% and PSlope did not vary considerably with any increase in the number of discretisation points.

Slope of Regression is particularly sensitive to changes in block size in the east-west direction in the well informed areas, with block sizes greater than 20 m giving increasingly poorer PSlope (Figure 14.9).

Both KE% and PSlope are not sensitive to block size in the north-south and vertical dimensions until relatively large block sizes are used. In the moderately informed areas, the KNA showed that small blocks in the east-west and vertical directions and larger blocks in the north-south direction are preferred. The KNA resulted in a choice of block size of 40 mN by 20 mE by 10 mZ.

The minimum number of composites used for an estimate was 10 and the maximum number 30 (Figure 14.10). Once more than 30 composites were used in a well-informed area, there was no significant increase in the Kriging Efficiency or Slope of Regression. Less than a discretisation level of 3N by 3E by 3Z resulted in a poorer estimate and little improvement was noted with a higher level of discretisation.

**Figure 14.9 Example of Block Size Selection Well Informed High Grade Area (left) and Moderately Informed Medium Grade Supergene Area (right)**

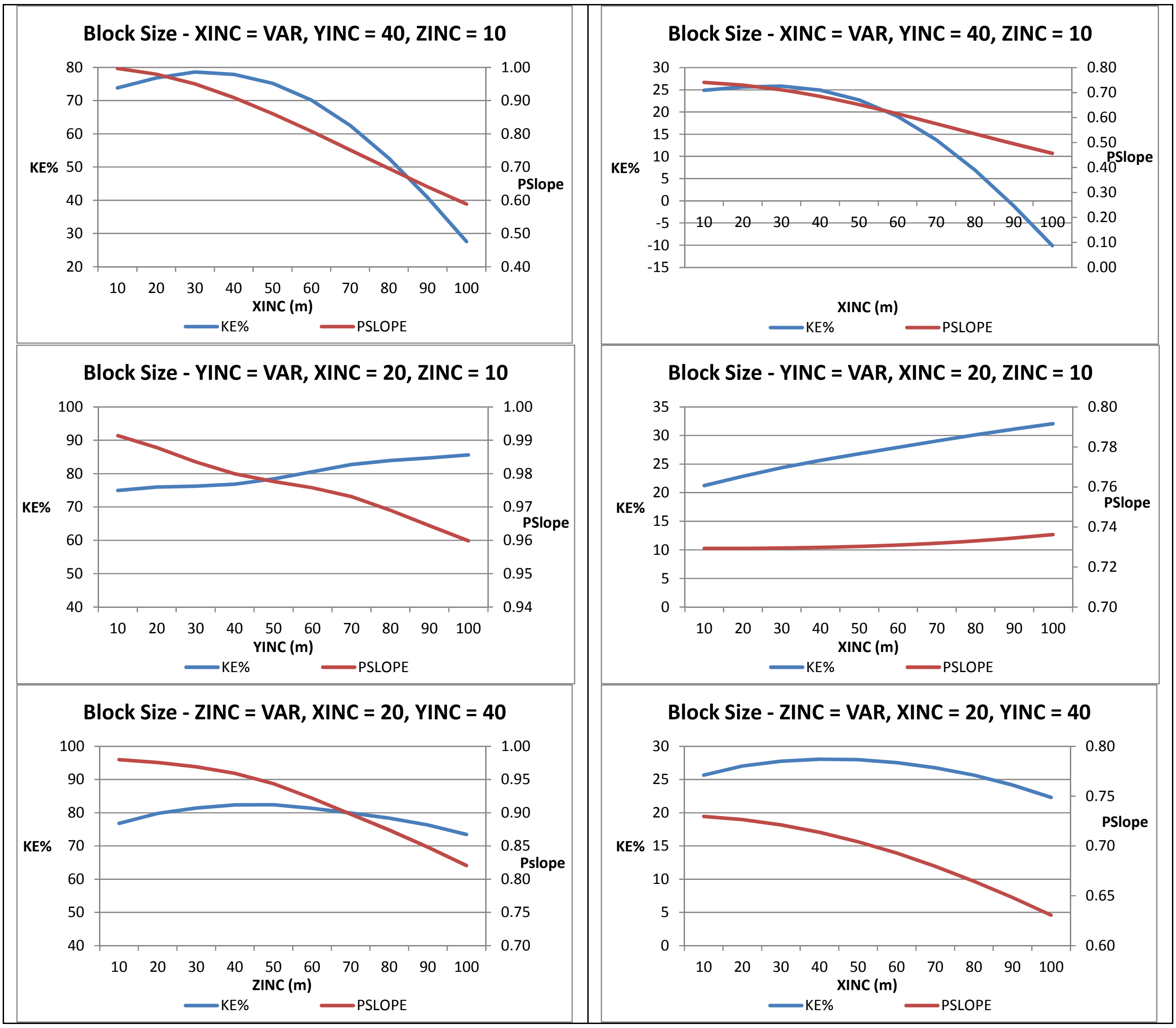
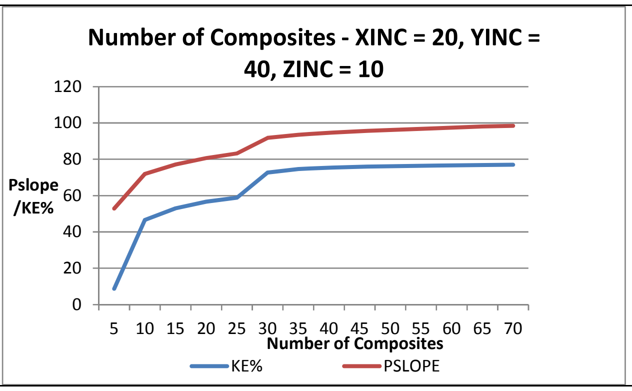
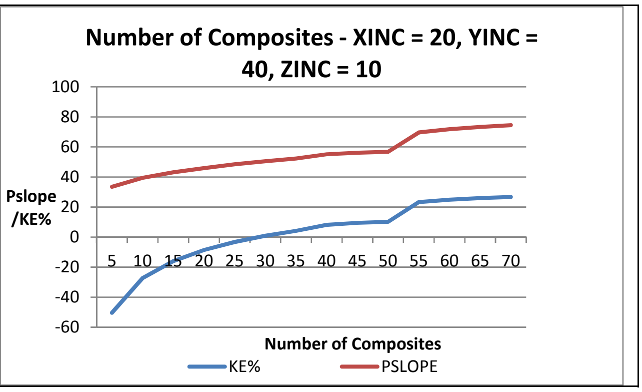


**Figure 14.10 Example of Number of Composite Selection - Well Informed High Grade Area (left) and Moderately Informed Medium Grade Supergene Area (right)**




## 14.8 Block Modelling

The block model prototype parameters are shown in Table 14.10. The cells were split to a minimum sub-cell of 2.5 mX by 5 mY by 2 mZ in order to well represent the wireframe model boundaries.

**Table 14.10 Block Model Prototype Parameters for Kitumba**

| XINC | YINC | ZINC | XMORIG | YMORIG | ZMORIG | NX | NY | NZ |
|---|---|---|---|---|---|---|---|---|
| 20 | 40 | 10 | 478700 | 8373240 | 760 | 59 | 33 | 79 |

Block models were created by filling below the topographic surface, above and below the leached/low Cu domain surface, above and below the supergene-hypogene interface and within the 0.30% and 4.0% CUPC grade shells. Block models were coded by mineralisation domain and grade shell so that estimation was controlled by the various zones.

## 14.9 Estimation

The following attributes were estimated:

- Total copper in percent (CUPC).
- Acid soluble copper in percent (CUASPC).
- Cobalt in parts per million (COPPM).
- Gold in grams per tonne (AUGT).
- Silver in grams per tonne (AGGT).

- Uranium in parts per million (UPPM).
- Iron in percent (FEPC).
- Sulphur in parts per million (SPPM).
- Manganese in parts per million (MNPPM).
- Relative Density (SG).

For CUPC, CUASPC, SPPM and SG a semi-soft boundary was used to select data between the leached, supergene and hypogene domains whereby data occurring up to 6 m below these interfaces was included in the supergene estimation data and vice-versa. For the grade shells a semi-soft boundary over 6 m was also used except for the 4% Cu grade shell where an 8 m interface both sides of the boundary was used. Direct observation of the drillhole data against the domain wireframes showed that the copper estimation domains and mineralogical state do not control the grade distribution of Co, Au, Ag, U and Fe and therefore the mineralogical state domains and grade shells were not used during estimation of these elements.

The search distance and the rotation angles that defined the search ellipsoids were set at the variogram range and rotations for each attribute and the number of composites used for estimation was guided by the KNA for CUPC. If an estimate was not achieved within the search ellipse volume, the search ellipse was expanded by 50%. Should an estimate still not be achieved, the search ellipse was further expanded so that all the cells within the area of interest were estimated. Discretisation was set at 3N by 3E by 3Z. Neither the octant search nor the MaxKey options in CAE Studio 3 were used. The search parameters are shown in Table 14.11.

Ordinary Kriging was used to estimate the attributes into the block model cells using parent cell estimation.

**Table 14.11 Search Parameters for Kitumba**

| | Search Distance | | | Search Angle | | | Rotation Axis | | | First Search Volume | | Second Search Volume | | | Third Search Volume | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Attribute** | **1** | **2** | **3** | **1** | **2** | **3** | **1** | **2** | **3** | **Min Num.** | **Max Num.** | **Factor** | **Min Num.** | **Max Num.** | **Factor** | **Min Num.** | **Max Num.** |
| **Low Cu/Leached Domain** | | | | | | | | | | | | | | | | | |
| CUPC | 220 | 110 | 145 | 90 | 75 | 155 | 3 | 1 | 3 | 10 | 30 | 1.5 | 10 | 30 | 12 | 10 | 30 |
| CUASPC | 220 | 110 | 145 | 90 | 75 | 155 | 3 | 1 | 3 | 10 | 30 | 1.5 | 10 | 30 | 12 | 10 | 30 |
| SG | 305 | 260 | 190 | 90 | 70 | 90 | 3 | 1 | 3 | 10 | 30 | 1.5 | 10 | 30 | 12 | 10 | 30 |
| SPPM | 75 | 120 | 100 | 90 | 90 | 90 | 3 | 1 | 3 | 10 | 30 | 1.5 | 10 | 30 | 12 | 10 | 30 |
| **Supergene Domain** | | | | | | | | | | | | | | | | | |
| CUPC | 130 | 360 | 60 | 90 | 75 | 90 | 3 | 1 | 3 | 10 | 30 | 1.5 | 10 | 30 | 1.5 | 10 | 30 |
| CUASPC | 130 | 360 | 60 | 90 | 75 | 90 | 3 | 1 | 3 | 10 | 30 | 1.5 | 10 | 30 | 1.5 | 10 | 30 |
| SG | 305 | 260 | 190 | 90 | 70 | 90 | 3 | 1 | 3 | 10 | 30 | 1.5 | 10 | 30 | 1.5 | 10 | 30 |
| SPPM | 320 | 480 | 160 | 90 | 75 | 90 | 3 | 1 | 3 | 10 | 30 | 1.5 | 10 | 30 | 1.5 | 10 | 30 |
| **Hypogene Domain** | | | | | | | | | | | | | | | | | |
| CUPC | 125 | 160 | 60 | 110 | 65 | 100 | 3 | 1 | 3 | 10 | 30 | 1.5 | 10 | 30 | 12 | 10 | 30 |
| CUASPC | 125 | 160 | 60 | 110 | 65 | 100 | 3 | 1 | 3 | 10 | 30 | 1.5 | 10 | 30 | 12 | 10 | 30 |
| SG | 305 | 260 | 190 | 90 | 70 | 90 | 3 | 1 | 3 | 10 | 30 | 1.5 | 10 | 30 | 12 | 10 | 30 |
| SPPM | 600 | 300 | 130 | 110 | 65 | 100 | 3 | 1 | 3 | 10 | 30 | 1.5 | 10 | 30 | 12 | 10 | 30 |
| **All Domains** | | | | | | | | | | | | | | | | | |
| COPPM | 160 | 220 | 110 | 90 | 70 | 90 | 3 | 1 | 3 | 10 | 30 | 1.5 | 10 | 30 | 12 | 10 | 30 |
| AUGT | 750 | 440 | 28 | 90 | 70 | 90 | 3 | 1 | 3 | 10 | 30 | 1.5 | 10 | 30 | 12 | 10 | 30 |
| AGGT | 550 | 650 | 500 | 90 | 70 | 90 | 3 | 1 | 3 | 10 | 30 | 1.5 | 10 | 30 | 12 | 10 | 30 |
| UPPM | 800 | 1600 | 170 | 90 | 70 | 90 | 3 | 1 | 3 | 10 | 30 | 1.5 | 10 | 30 | 12 | 10 | 30 |
| FEPC | 520 | 650 | 280 | 90 | 70 | 90 | 3 | 1 | 3 | 10 | 30 | 1.5 | 10 | 30 | 12 | 10 | 30 |
| MNPPM | 260 | 300 | 200 | 90 | 70 | 90 | 3 | 1 | 3 | 10 | 30 | 1.5 | 10 | 30 | 12 | 10 | 30 |

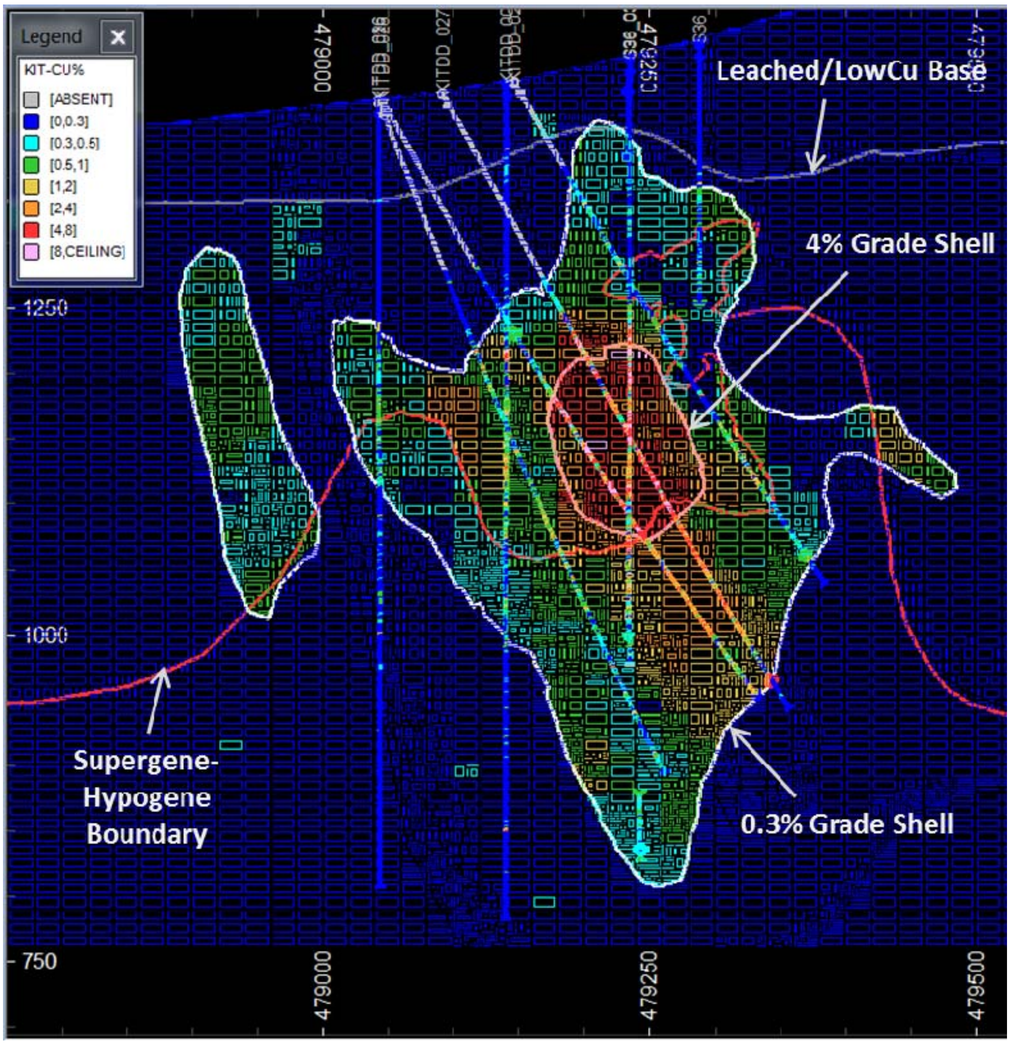
### 14.9.2 Validation of the Estimates

The models were validated by:

- Visual examination of the input data against the block model estimates.
- Sectional validation.
- Comparison of the de-clustered input data statistics against the model statistics.

The block model was examined visually in sections to ensure that the drillhole grades were locally well represented by the model. Areas of excessive grade spreading were checked for. The model visually validated well against the data (Figure 14.11).

**Figure 14.11 E-W Section# Along 8373880 Showing Block Model CUPC Values vs Drillhole Values**

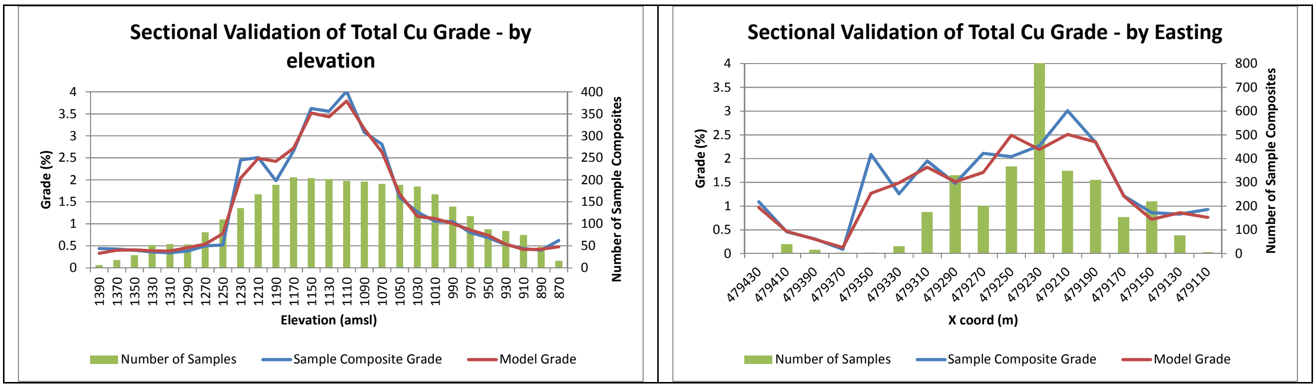



*# View shows data in a 20 m corridor either side of the section.*

Scale can be determined from the grid.

Sectional validation plots were constructed for CUPC, in order to compare the average grades of the block model against the input data along a number of corridors in various directions through the deposit. Sectional validation plots for CUPC are shown in Figure 14.12. These show that the estimates are smoothed yet retain the grade trends across the deposit.

**Figure 14.12 Sectional Validation Plots for CUPC**




Once the classification was completed (refer to later in this report), the de-clustered mean of the 2 m composite data within the areas classified as Measured and Indicated was compared to the Measured and Indicated model mean Table 14.12. The global comparison is good, there being a slight tendency for the model mean to be lower than the data mean. The difference is considered to be caused by the generally lower grade areas being large and the least drilled and not fully compensated by the small de-clustering cell size.

**Table 14.12 Comparison Between Drillhole and Model Data Values (M&I Area)**

| Domain / Variable | Mean Model (M&I) | Mean Data |
|---|---|---|
| CUPC | 1.00 | 1.07 |
| CUASPC | 0.37 | 0.39 |
| COPPM | 156 | 162 |
| AUGT | 0.03 | 0.04 |
| AGGT | 0.81 | 0.85 |
| UPPM | 28 | 30 |
| FEPC | 14.0 | 13.9 |
| MNPPM | 5,560 | 5,610 |
| SPPM | 12,155 | 12,801 |
| SG | 2.67 | 2.67 |

### 14.9.3 Geological Losses

A geological loss factor was not applied to the estimate as the type of structure and massive style of mineralisation does not warrant any losses to the Mineral Resource. There are a number of faults in the Mineral Resource, but these are considered to be a control on the mineralisation rather than constituting a loss.

## 14.10 Classification

Classification of the Kitumba Mineral Resource was based on confidence in the data, confidence in the geological model and the continuity of grade. The main considerations in the classification of the Kitumba Mineral Resource were as follows.

The majority of the data that informs the Mineral Resource has been collected by Blackthorn in the Phase 6 and Phase 7 drilling programmes conducted between 2012 and 2014. Included with the data are a number of drillholes that were collected by BHP Billiton when a joint venture between Blackthorn and BHP Billiton was in place. The data were collected using documented 'best practice' principals and were subject to external laboratory QAQC. The pre-Phase 6 drillhole cores have been closely examined and re-logged according to lithological and mineralogical codes consistent with Blackthorn's current understanding of the deposit. The data are stored in a validated relational database and can be considered to be of high confidence.

The geological framework hosting the mineralisation is well understood and mineralisation domains have been defined based on logged mineralisation state and grades. Grade trends within the deposit are predictable as confirmed by the infill drilling programme that validated the gross shape and grade of the previous estimate. High local geological variability in terms of degree of brecciation and detailed lithology does occur due to the nature of the deposit, it being a supergene enriched brecciated body. However, the short scale geological variability does not impact on the grade distribution on a bulk mining scale and the close drillhole spacing is sufficient to understand the grade distribution at the envisaged scale of mining.

The variogram ranges for copper are well informed by the close spaced drilling in the core of the deposit. Grade trends for the different metals are well understood and fit well with the geological understanding of the deposit controls. The drilling is sufficient to confirm grade continuity between the drillholes in the well and moderately drilled areas, the drillhole spacing being largely well within the variogram range.

The estimate was classified on the following basis, while taking into account the aforementioned factors:

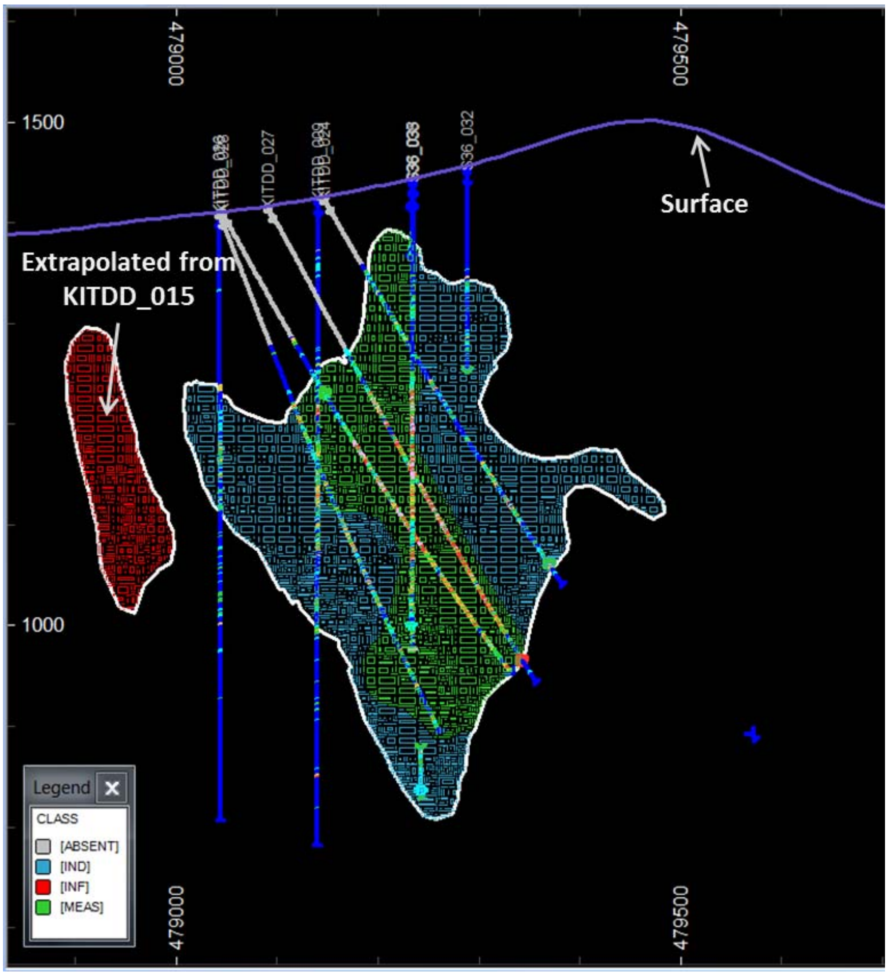
Mineralisation was considered as Measured Resources where the Kriging Efficiency was greater than >0.5 and the Slope of Regression was generally more than 80%. Where the zones of high Kriging Efficiency and Slope of Regression were not connected, these were classified as Indicated, a contiguous high confidence zone informed by several drillholes being required for a Measured Resource.

Indicated Resources were considered when an estimate was achieved with a minimum 20 composites sourced by the first search volume, i.e., all composites were within the variogram range. These estimates were classified as Indicated Mineral Resources when in the view of the Competent Person that the confidence in the grade shell interpretation was good. Indicated Mineral Resources were limited to a maximum of 60 m along strike from last line of drillholes.

The remainder of the mineralisation within the >0.30% grade shell was classified as Inferred Mineral Resources.

The estimate is influenced by the interpretation of mineralisation and grade domains. In the area classified as a Measured Resource, the drillhole intersection points used for the interpretation of these domains are mostly between 20 m and 40 m apart and the interpretation is considered robust (Figure 14.13). In the area classified as Indicated Resources the control points are further apart (mostly between 80 m and 120 m apart) and the confidence in the geological interpretation is lower and therefore significant changes to local estimates may occur.

**Figure 14.13 E-W Section# Along 8373880 Showing Mineral Resource Classification in a Well Drilled Area**




*# View shows data in a 20 m corridor either side of the section.*

Scale can be determined from the grid.

The majority of the Inferred Resource at Kitumba comprises an area to the west of the main zone of mineralisation and the northern and southern extremities. This western area was estimated as a result of extrapolation along the trend of mineralisation from one drillhole (KITDD_015) and no drilling to the north and south along the mineralisation trend has been completed. The Inferred Resource has been extrapolated by a maximum of 140 m along strike from the nearest drillhole.

The down dip extents are defined by the 0.30% Cu threshold interpreted from the drillhole data and therefore for most of the deposit no extrapolation occurs at depth. A limited amount of down dip extrapolation to approximately 150 m from the nearest drillhole occurred in the southern extremity of the Mineral Resource area, which was guided by the results of the Indicator Kriging used to define the 0.30% Cu grade shell. In the east-west direction the Mineral Resource extent has been well defined by the drilling grid with extrapolation being limited by the short search distance used in the Indicator Kriging used to define the 0.30% Cu grade shell. The maximum distance of extrapolation in the east-west direction occurs toward the northern and southern extremities, the extrapolation distance being less than approximately 50 m. The Inferred Mineral Resources above a cut-off grade of 0.5% total Cu all occur within the modelled 0.30% Cu grade shell.

The close drillhole spacing in the area classified as a Measured Resource is sufficient so that any variation in the estimate of the Measured Resource area due to additional data will be unlikely to significantly affect total economic viability. Despite the lower confidence in the Indicated area, the deposit is sufficiently well understood so that any changes are not expected to significantly change the total quantity and quality of the Indicated Mineral Resource. The nature, quality, amount and distribution of the data in the Measured and Indicated areas allows for confident determination of the geological framework and to assume continuity of mineralisation allowing for the application of Modifying Factors within a Technical and Economic Study to at least Pre-feasibility level as defined by CIM (2014).

The Inferred Resources that are derived from extrapolation outside of the drillhole grid along the mineralisation trends (as defined by the variography), or informed by sparse drilling, are considered to be high risk estimates that may change significantly with additional data. It cannot be assumed that all or part of an Inferred Mineral Resource will necessarily be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration.

The Mineral Resource estimate and classification appropriately reflects the Qualified Person’s view of the deposit.

## 14.11 Mineral Resource Statement

The Mineral Resource estimate has been completed by Mr. J.C. Witley (BSc Hons) who is a geologist with 26 years’ experience in base and precious metals exploration and mining as well as Mineral Resource evaluation and reporting. He is a Principal Resource Consultant for The MSA Group (an independent consulting company), is a member in good standing with the South African Council for Natural Scientific Professions (SACNASP) and is a Member of the Geological Society of South Africa (GSSA). Mr. Witley has the appropriate relevant qualifications and experience to be considered a ‘Qualified Person’ for the style and type of mineralization and activity being undertaken as defined in National Instrument 43-101 Standards of Disclosure of Mineral Projects (NI 43-101).

The Mineral Resource was classified into Measured, Indicated or Inferred categories in accordance with the guidelines of the 2012 edition of the JORC Code. It should be noted that there are no material differences between the Mineral Resource categories reported herein whether using those defined by JORC (2012) or the CIM Definition Standards on Mineral Resources and Reserves (CIM Definition Standards) adopted by CIM Council on May 10, 2014. The Mineral Resource, Mineral

Reserve, and Mining Study definitions as described in the CIM Definition Standards are incorporated, by reference, into National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101).

The Kitumba Mineral Resource is reported above a base case cut-off grade of 1.0% Cu. It should be noted that the cut-off grades applied are not the result of detailed economic analysis and therefore Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

There are no known legal, political, environmental, or other risks that could materially affect the potential development of the Mineral Resources.

The maximum depth of the reported Mineral Resource at a 1% Cu cut-off grade is at an RL of 770 m, which is approximately 660 m below surface. No Mineral Resources occur above the stated 1.0% Cu cut-off-grade within 150 m of surface, however at a 0.5% Cu cut-off the mineralisation occurs from an average of approximately 100 m below surface.

At a cut-off-grade of 1.0% Cu, the total Measured and Indicated Mineral Resource is 34.7 million tonnes at a total copper grade of 2.29% (Table 14.13). This equates to 0.8 million tonnes of copper in-situ.

**Table 14.13 Kitumba Mineral Resource[#] Above a Cut-off Grade of 1.0% Cu, as at 5 December 2013**

| Category | Tonnes (Millions) | Cu % | Acid Soluble Cu % | Co ppm | Au g/t | Ag g/t | U ppm | Density $t/m^3$ |
|---|---|---|---|---|---|---|---|---|
| **Supergene Domain** | | | | | | | | |
| Measured | 6.1 | 3.44 | 1.66 | 205 | 0.04 | 1.3 | 25 | 2.51 |
| Indicated | 15.2 | 2.07 | 1.00 | 180 | 0.03 | 0.9 | 26 | 2.60 |
| M&I | 21.3 | 2.46 | 1.19 | 187 | 0.03 | 1.0 | 26 | 2.57 |
| Inferred | 0.2 | 1.12 | 0.28 | 124 | 0.16 | 0.4 | 22 | 2.66 |
| **Hypogene Domain** | | | | | | | | |
| Measured | 4.4 | 2.23 | 0.45 | 247 | 0.04 | 1.0 | 21 | 2.86 |
| Indicated | 9.0 | 1.93 | 0.57 | 210 | 0.03 | 0.9 | 32 | 2.83 |
| M&I | 13.4 | 2.03 | 0.53 | 222 | 0.03 | 0.9 | 28 | 2.84 |
| Inferred | 3.9 | 1.39 | 0.23 | 415 | 0.02 | 0.7 | 31 | 2.81 |
| **Combined Domain** | | | | | | | | |
| Measured | 10.5 | 2.93 | 1.15 | 223 | 0.04 | 1.2 | 23 | 2.66 |
| Indicated | 24.2 | 2.02 | 0.84 | 191 | 0.03 | 0.9 | 28 | 2.69 |
| M&I | 34.7 | 2.29 | 0.93 | 201 | 0.03 | 1.0 | 27 | 2.67 |
| Inferred | 4.1 | 1.38 | 0.23 | 401 | 0.03 | 0.7 | 31 | 2.80 |

\# ***All tabulated data have been rounded and therefore minor computational errors may occur.***

In order to illustrate the sensitivity of the Mineral Resource to cut-off grade, the Mineral Resource is tabulated using a number of cut-off grades in Table 14.14 for Measured and Indicated (M&I) Resources and Table 14.15 for Inferred Resources.

**Table 14.14 Kitumba Measured and Indicated Mineral Resource# by Cut-Off Grade, as at 5 December 2013**

| Cut Off Grade (Cu%) | Tonnes (Millions) | Cu % | Acid Soluble Cu % | Co ppm | Au g/t | Ag g/t | U ppm | Density $t/m^3$ |
|---|---|---|---|---|---|---|---|---|
| 0.50 | 81.6 | 1.37 | 0.52 | 170 | 0.04 | 0.9 | 28 | 2.67 |
| 1.00 | 34.7 | 2.29 | 0.93 | 201 | 0.03 | 1.0 | 27 | 2.67 |
| 1.40 | 25.1 | 2.72 | 1.16 | 208 | 0.03 | 1.0 | 27 | 2.65 |

***# All tabulated data have been rounded and therefore minor computational errors may occur.***

**Table 14.15 Kitumba Inferred Mineral Resource# by Cut-Off Grade, as at 5 December 2013**

| Cut Off Grade (Cu%) | Tonnes (Millions) | Cu % | Acid Soluble Cu % | Co ppm | Au g/t | Ag g/t | U ppm | Density $t/m^3$ |
|---|---|---|---|---|---|---|---|---|
| 0.50 | 26.2 | 0.79 | 0.15 | 175 | 0.04 | 0.6 | 26 | 2.71 |
| 1.00 | 4.1 | 1.37 | 0.23 | 400 | 0.03 | 0.7 | 30 | 2.80 |
| 1.40 | 1.4 | 1.85 | 0.28 | 231 | 0.03 | 0.5 | 23 | 3.00 |

***# All tabulated data have been rounded and therefore minor computational errors may occur.***

## 14.12 Comparison with Previous Estimates

Mining study work completed since the April 2013 Mineral Resource Estimate indicate that the mining project is more likely to be an underground project rather than an open pit. Accordingly, the base case cut-off grade was increased from the 0.5% total copper used in April 2013 to 1.0% total copper. The April 2013 estimates have been re-stated accordingly in Table 14.16 for comparison purposes.

The December 2013 estimate shows an increase in the tonnes and copper grade over the April 2013 estimate. This was primarily the result of the higher grade infill drilling data collected during the Phase 7 programme and therefore the increase was primarily in the Measured and Indicated (M&I) area.

**Table 14.16 Comparison of Kitumba Mineral Resource# Above a Cut-off Grade of 1.0% Cu 05 December 2013 vs 08 April 2013**

| Category | Tonnes (Millions) | Cu % | Au g/t | Ag g/t | U ppm | Density t/m$^3$ | Tonnes (Millions) | Cu % | Au g/t | Ag g/t | U ppm | Density t/m$^3$ |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | December 2013 | | | | | | April 2013* | | | | | |
| Measured | 10.4 | 2.93 | 0.04 | 1.2 | 23 | 2.65 | - | - | - | - | - | - |
| Indicated | 24.2 | 2.02 | 0.03 | 0.9 | 28 | 2.68 | 29.8 | 2.13 | 0.03 | 1.0 | 27 | 2.67 |
| M&I | 34.7 | 2.29 | 0.03 | 1.0 | 27 | 2.67 | 29.8 | 2.13 | 0.03 | 1.0 | 27 | 2.67 |
| Inferred | 4.1 | 1.37 | 0.03 | 0.7 | 30 | 2.80 | 3.5 | 1.39 | 0.04 | 0.4 | 21 | 2.95 |

# ***All tabulated data have been rounded to one decimal place for tonnage and to either no or two decimal places for grades***

* ***(Source, MSA 2013)***

At a 0.5% Cu cut-off grade, a 7% decrease in the M&I Mineral Resource tonnes and a 17% increase in copper grade occurred between the April 2013 and December 2013 estimates, resulting in 9% more contained copper (Table 14.17). Gold and uranium values remained almost the same overall.

**Table 14.17 Comparison of Kitumba Mineral Resource# Above a Cut-off Grade of 0.5% Cu 05 December 2013 vs 08 April 2013**

| Category | Tonnes (Millions) | Cu % | Au g/t | Ag g/t | U ppm | Density t/m$^3$ | Tonnes (Millions) | Cu % | Au g/t | Ag g/t | U ppm | Density t/m$^3$ |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | December 2013 | | | | | | April 2013* | | | | | |
| Measured | 14.6 | 2.30 | 0.04 | 1.0 | 23 | 2.67 | - | - | - | - | - | - |
| Indicated | 67.0 | 1.16 | 0.03 | 0.8 | 29 | 2.67 | 87.6 | 1.17 | 0.03 | 0.9 | 29 | 2.67 |
| M&I | 81.6 | 1.37 | 0.04 | 0.9 | 28 | 267 | 87.6 | 1.17 | 0.03 | 0.9 | 29 | 2.67 |
| Inferred | 26.2 | 0.79 | 0.04 | 0.6 | 26 | 2.71 | 21.3 | 0.77 | 0.05 | 0.5 | 20 | 2.70 |

# ***All tabulated data have been rounded to one decimal place for tonnage and to either no or two decimal places for grades.***

* ***(Source, MSA 2013)***

## 14.13 Summary and Recommendations

A number of opportunities have arisen from the drilling program and subsequent Mineral Resource estimate:

- A large portion of the Inferred Mineral Resource at Kitumba comprises an area to the west of the main zone of mineralisation. This area was estimated as a result of extrapolation along the trend of mineralisation from one drillhole (KITDD_015). As such a large proportion of the Inferred Mineral Resource at Kitumba carries significant risk and should be treated with caution. It is recommended that a number of holes are drilled in the western extension in order to confirm the nature and extent of this mineralisation.

- Should a larger amount of Measured Mineral Resource be required, this is likely to be achieved by drilling at a 40 m or closer spaced grid along strike to the north and south of the current well drilled area, which would de-risk the project further.

- The nature, quality, amount and distribution of the data in the Measured and Indicated areas allows for confident determination of the geological framework and to assume continuity of mineralisation allowing for the application of Modifying Factors within a Technical and Economic Study to at least Pre-feasibility level as defined by CIM (2014).

The Phase 7 drilling was successful in upgrading the confidence of a significant portion of the Mineral Resource from Indicated to Measured and increasing the contained copper content of the project. Furthermore, the high grade core to the Kitumba deposit was confirmed by a number of drillholes and the confidence in the positions of the mineralisation state boundaries was enhanced.

# 15.0 MINERAL RESERVE ESTIMATES

The mining study was undertaken by AMC Consultants Pty Ltd (AMC) at pre-feasibility study level and is based on the Mineral Resources discussed in Section 14.

The Project Mineral Reserve estimate, classified and reported in accordance with the Canadian Securities Administrators National Instrument 43-101 (NI 43-101) and the corresponding CIM Definition Standards on Mineral Resources and Mineral Reserves, is listed in Table 15.1.

**Table 15.1 Kitumba Mineral Reserve Estimate**

| Item | Tonnes (Mt) | Grade (% Cu) | Metal (kt Cu) |
|---|---|---|---|
| Proven Mineral Reserve | 11.9 | 2.44 | 291 |
| Probable Mineral Reserve | 19.6 | 1.79 | 350 |
| Total Mineral Reserve | 31.5 | 2.04 | 641 |

Mineral Reserves are defined within an underground mine plan generated considering diluted Measured and Indicated Mineral Resources.

Mineral Resources were converted to Mineral Reserves recognizing the level of confidence in the Mineral Resource estimate and reflecting any modifying factors. The Proven Mineral Reserve is based on Measured Mineral Resources, and the Probable Mineral Reserve is based on Indicated Mineral Resources, after consideration of all mining, metallurgical, social, environmental, statutory and financial aspects of the Project.

The Mineral Reserve is that part of the Mineral Resource which can be economically mined by underground mining methods. Dilution of the Mineral Resource model and an allowance for ore loss was included in the Mineral Reserve estimate.

The key non-mining modifying factors used in generating the Mineral Reserves are listed in Table 15.2.

**Table 15.2 Non-mining Modifying Factors**

| Item | Unit | Value |
|---|---|---|
| Processing Cost | $/t | 42.51 |
| General and administration cost | $/t | 1.67 |
| Metallurgical recovery | % | 90 |
| Copper price | $/lb | 3.50 |

The key modifying factors used in generating the Mineral Reserves are listed in Table 15.3.

**Table 15.3 Mining Modifying Factors**

| Item | Unit | Value |
|---|---|---|
| Mining dilution - development | % | 0 |
| Mining recovery - development | % | 100 |
| Mining dilution - caving[1] | % | 27 |
| Mining recovery - caving[1] | % | 77 |
| Cut-off grade - design | %Cu | 1 |

[1] **As estimated by AMC's SLC mixing algorithm.**

Sub-level Cave (SLC) inventories were estimated using an SLC mixing algorithm which is based on draw curves complied from the results of draw maker trials and detailed in Power (2004). The algorithm estimates recovery and dilution for production from sub-level caving operation.

The algorithm estimates drawn production by mixing material from different sources including:

- Primary material – material sourced from the ring fired at the draw point.
- Secondary material – material sourced from rings fired on the level immediately above the draw point.
- Tertiary material – material sourced from rings fired on 2 levels or above the draw point.
- External material – dilution material from the cave.

The algorithm assumes all material flow is vertical.

AMC is not aware of any mining, metallurgical, infrastructure, permitting, and other relevant factors which could materially affect the Mineral Reserve estimates.

# 16.0 MINING METHODS

## 16.1 Introduction

The mining study for the project considered Measured Mineral Resources, Indicated Minerals Resources and Inferred Mineral Resources.

The mining study considered SLC, a combination of SLC and sub-level open stoping (SLOS) and block caving mining methods. Previous studies considered open pit mining and SLOS.

## 16.2 Mining Method

SLC was selected as the preferred and most suitable mining method for the project.

The factors influencing the mining method selection were:

- The massive geometry of the deposit.
- Near surface mineralisation has been leached of copper.
- A geotechnical assessment that indicates that caving can be induced in areas overlying the orebody.
- Technology – SLC is a mechanized method that allows a high production rate, and low cost.
- Safety – SLC is a non-entry method, where personnel do not enter stoping excavations.

SLC is a top down mining method allowing early production with relatively little predevelopment. In-situ ore is progressively blasted in horizontal slices. Rock from above the production horizon caves into the void created by ore extraction, and ore is diluted by the caved rock. Dilution is managed by draw control. As the process progresses, caving propagates upwards and can create surface subsidence.

To manage dilution cross-cuts have been conservatively designed to allow effective draw control by including; a 25 m sub-level spacing, a 14 m centre to centre spacing and transverse extraction.

## 16.3 Geotechnical

### 16.3.1 Regional Stress State

The stress regime at the project was estimated from a number of published sources for the region. The stress tensor data as used in the numerical modelling are presented in Table 16.1.

**Table 16.1 Stress Data Used as Input Parameters for Numerical Modelling**

| Stress Component | Stress Orientation (Azimuth/Plunge) (º) | Stress Magnitude (MPa) |
|---|---|---|
| $\sigma_1$ | 225/65 | 0.0419z+7.8 |
| $\sigma_2$ | 315/00 | 0.0336z+4 |
| $\sigma_3$ | 045/25 | 0.028z |

### 16.3.2 Intact Rock Properties

The summary of unconfined compressive strength (UCS) and elastic properties (Young's modulus and Poisson's ratio) for each major rock type, based on testwork, is presented in Table 16.2.

**Table 16.2 Summary of intact Rock Strength and Elastic Properties of Main Rock Types**

| Rock Type | Code | UCS, MPa | E, GPa | ν |
|---|---|---|---|---|
| Quartz-feldspar porphyry granite | QFPG | 98 | 51 | 0.22 |
| Feldspar porphyry syenite | FPSY | 76 | 32 | 0.32 |
| Polymictic breccia | BREC | 105 | 54 | 0.25 |
| Iron oxide replaced | IORE | 117 | 38 | 0.21 |

### 16.3.3 Rock Mass Characterisation

AMC utilized two widely recognized rock mass classification systems used in underground mining: Q index, which was utilised for ground support selection (Barton, Lien and Lunde, 1974) and MRMR system, typically used for caveability assessment (Laubscher, 1990). The Q index for rock mass around capital development drives was calculated to be in the range from 0.7 to 13.3, which is rated as ‘Very Poor’ to ‘Good’, while the MRMR index for the rock mass above the production area (caving zone) ranged between 25 and 42 (‘Poor’ to ‘Fair’).

### 16.3.4 Ground Support

The analysis indicates that 2.4 m long rock bolts installed on a systematic pattern with nominal spacing of 1.3 m to 1.7 m (depending on the type of surface support) will provide safe ground conditions. In capital development fibre-reinforced shotcrete is recommended as surface support at a nominal thickness of 60 mm. Galvanised weld mesh is sufficient in the ore drives due to the short life of excavations.

Routine cable-bolting of development intersections is not required due to the low stress environment and absence of large wedges as indicated by the kinematic analysis. However, this is to be confirmed by the detailed structural mapping and subsequent wedge analysis as soon as the development commences.

### 16.3.5 Numerical Modelling

Mining induced stress analysis was carried out using Map3D boundary element method to simulate the stress distributions around excavations in an elastic homogeneous rock mass.

The modelling results indicate that both declines and other capital development drives, including cross cuts, are located at an adequate stand-off distance from the production area. Stress levels in the pillars between the ore drives are expected to be relatively high with some possible stress spalling in the top right corner of the ore drives. Due to the short life of the ore drives these conditions should be adequately managed by the ground support regime recommended above. The stress magnitude in the backs of ore drives is not high enough to cause production blast holes to squeeze. However, some shearing may occur in the holes closest to the cave due to the increased shear stress on the sub horizontal joint set.

### 16.3.6 Caveability

The ability of the rock mass to cave was assessed on a preliminary basis using a method proposed by Laubscher (1990). This relates the MRMR index to the size of the undercut, expressed in terms of the hydraulic radius (HR). The HR for a given undercut shape (in plan) is simply the area divided by the perimeter.

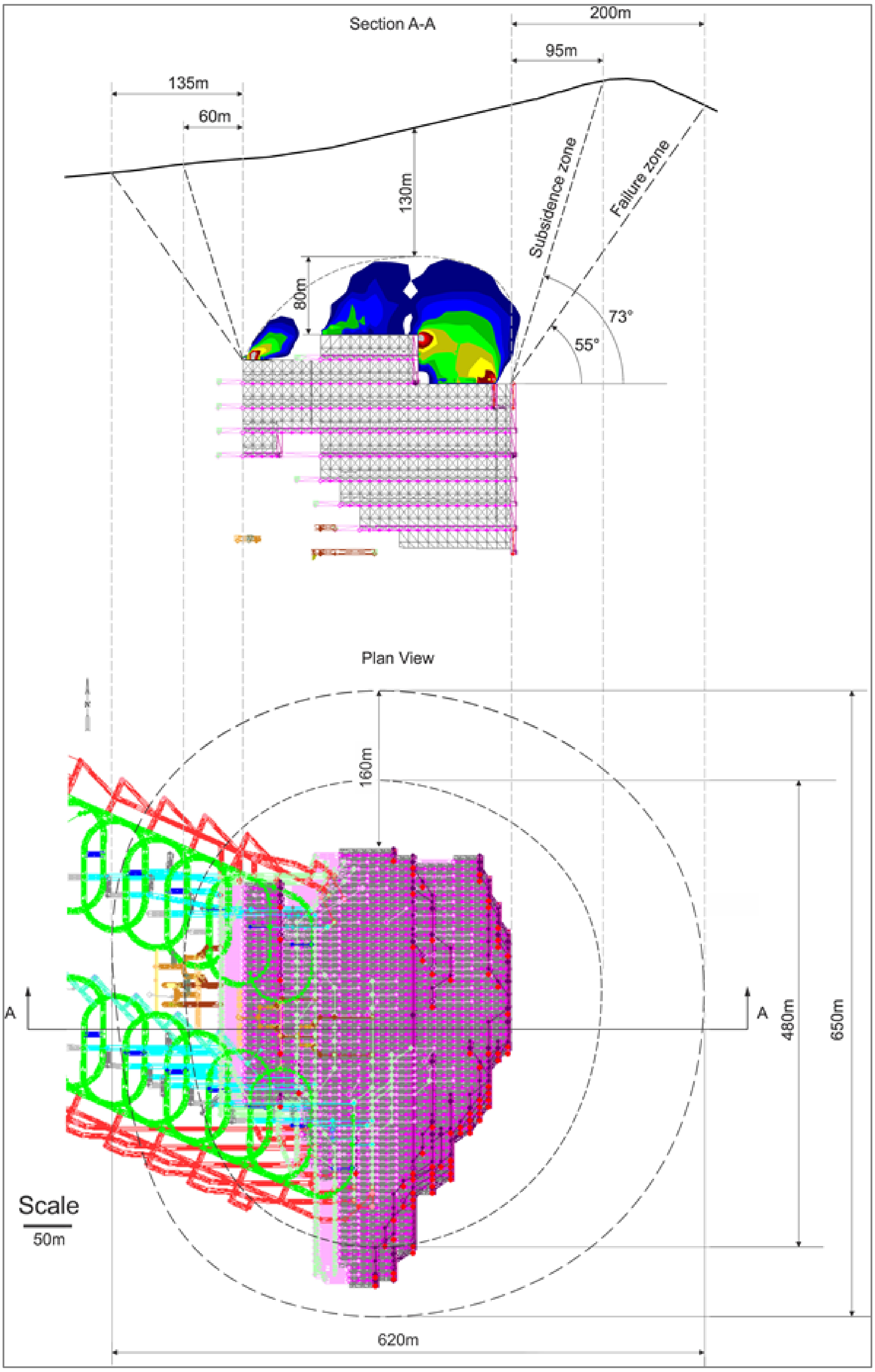
The analysis showed that the rock mass above the production zone will readily cave. The HR of the undercut is approximately 30 m, based on the currently planned dimensions of 80 m east-west and 250 m north-south. The maximum MRMR value of 42 (a conservative approach) was used in the analysis.

Shearing forces will likely be acting along discontinuities seeing that the major principal stress (225º/65º) plunges at a 38º angle relative to the dominant joint set (059º/27º). This will result in weakening the rock mass above the production zone, thus promoting caving.

### 16.3.7 Surface Subsidence

Surface subsidence typically occurs at steep angles (70° to 90°) projected from the perimeter of the cave up to the surface. The subsidence zone is surrounded by a failure zone which is projected at flatter angles from the cave. The cave angle at the project was determined as 73° based on an empirical analysis proposed by Laubscher (2000). This may result in surface subsidence zone above the production area up to 480 m in diameter. The angle of the failure zone was found to be 55° resulting in an approximately 650 m diameter failure zone, within which no mine infrastructure is to be built. Figure 16.1 illustrates the estimated failure zone and surface subsidence.

**Figure 16.1 Failure Zone and Surface Subsidence Around the Cave**



## 16.4 Cut-off Grade

The method used to select the cut-off grade (COG) for use as the basis of mine design was to compare different inventories derived from a range of COGs. For a given inventory, and its spatial distribution, assumptions are made about potential productivity and costs (drawing from AMC benchmarking). Simple schedules are prepared in Microsoft Excel. Conclusions were then drawn about the most favourable COG and production rate for the mine design.

AMC estimated fully and marginally costed COGs for a 3 Mtpa SLC project, for the application in the design and schedule.

The marginally costed – materials handling only – COG includes only mining materials handling costs and is the basis for the shut-off grade (SOG) applied in the SLC mixing algorithm.

The marginally costed – nil-mining – COG excludes mining costs and is used to define ore and waste of broken material that must be hauled to surface. This COG is applied during scheduling and predominately affects development material.

Table 16.3 lists COGs estimated for the OPFS.

**Table 16.3 Cut-off grades**

| Item | Unit | Fully Costed | Marginally Costed – Materials Handling Only | Marginally Costed – Nil Mining |
|---|---|---|---|---|
| Metallurgical recovery | % | 90 | 90 | 90 |
| Copper price | $/lb | 3.50 | 3.50 | 3.50 |
| Metal royalty | % | 6 | 6 | 6 |
| Effective price | $/lb | 3.29 | 3.29 | 3.29 |
| Mining cost | $/t | 40 | 15 | – |
| Processing cost | $/t | 30 | 30 | 30 |
| Total operating cost | $/t | 70 | 45 | 30 |
| Breakeven COG | % Cu | 1.00 | 0.7 | 0.45 |

## 16.5 Mine Design

***Boxcut***

Included in the mine design is a boxcut. The boxcut is a shallow open pit, mined for the purposes of accessing less-weathered and more competent ground conditions to commence decline development in (the mine portals).

The parameters used for boxcut design are:

- Depth: 50 m.

- Ramp gradient: 1:7.
- Face angle: 60º.
- Bench height: 10 m, final bench height: 15 m (to allow 8 m pillar between berm and portal backs).
- Berm width: 5 m.

***Mine Development***

The development design includes allowances for mine access, ventilation, infrastructure, materials handling and ore production.

The development design is based on:

- 25 m floor-to-floor sub-level intervals.
- Declines developed at 1:7 down.
- A minimum 50 m stand-off distance of capital infrastructure from the cave.
- A subsidence angle of 45 degrees.
- Decline curve radius of approximately 32 m.
- Cross-cut development designed for the transverse (east to west) extraction of the Mineral Reserves.

The development design includes:

- Twin declines, each to be used in a one-way configuration to reduce truck haulage congestion.
- Drives connecting the two declines provide for a secondary means of egress from the mine.
- The positioning of the declines also allows for access drives to be developed towards the extremities (towards the strike extents) of the Mineral Reserves.
- Footwall drives are developed along the strike extent of the orebody to access ore cross-cuts. Where possible, ore pass access development is located centrally on footwall drives.
- Return airway development is positioned at the extremities of the declines, and connected to the footwall drives so that air can be exhausted directly from production areas.

- Fresh airway development is included centrally.

- A transverse SLC layout, with production retreating from east to west.

- Ore cross-cuts are developed throughout the Mineral Resources on 14 m centres. Cross-cuts are 6 m wide and are separated by 8 m pillars between them.

- The mine is designed to reach a depth of approximately 450 m below surface, with production horizons developed over a 300 m vertical range.

The development designed makes allowance for materials handling by truck haulage and includes:

- The declines are positioned so that truck loading can be undertaken on a connecting drive situated between them.

- The connecting drive (between declines) intersects ore passes from sub-levels above.

- Ore passes can be fed at the top by loaders drawing ore from the cave on sub-levels above.

- Subsequently, loaders working the bottom of ore passes, load trucks in the drive connecting the two declines.

For the trucks, declines will be one way, so that passing is not required and congestion is greatly reduced.

Table 16.4 lists lateral development dimensions and metres in the design.

**Table 16.4 Lateral Development**

| Lateral Development | Dimensions (mW x mH) | Metres (m) |
|---|---|---|
| Decline | 5.5 x 6.0 | 5,906 |
| Decline stockpile | 5.5 x 5.5 | 390 |
| Return airway | 5.0 x 5.0 | 3,383 |
| Fresh airway | 5.0 x 5.0 | 600 |
| Service development | 5.0 x 5.0 | 260 |
| Materials handling | 5.5 x 5.5 | 1,234 |
| Level access | 5.5 x 5.5 | 1,869 |
| Level stockpile | 5.5 x5.5 | 388 |
| Footwall drive | 5.0 x 5.0 | 4,153 |
| Ore cross-cut | 4.5 x 6.0 | 40,163 |
| Slot drive | 5.0 x 5.0 | 4,738 |
| **Total** | **–** | **63,096** |

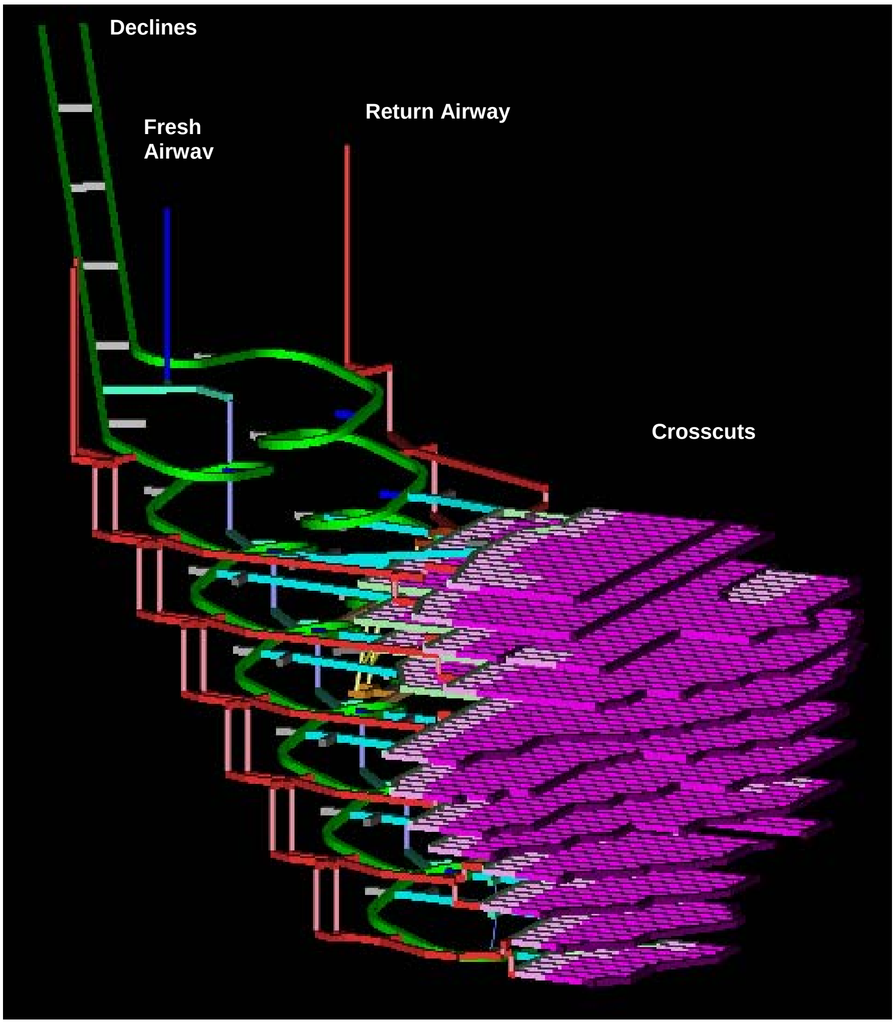
Table 16.5 lists vertical development dimensions and metres in the design.

**Table 16.5 Vertical Development**

| Vertical Development | Dimensions (m Diameter) | Metres (m) |
|---|---|---|
| Return air shaft | 4.5 | 437 |
| Return air rise | 4.5 | 927 |
| Fresh air shaft | 4.5 | 126 |
| Fresh air rise | 4.5 | 327 |
| Ore pass | 3.5 | 462 |
| Finger raise | 3.5 | 141 |
| Slot rise | 1 | 2,500 |
| **Total** | | **4,921** |

Figure 16.2 illustrates an isometric view of mine development.

**Figure 16.2 Isometric View of Mine Development**




### *Production*

The production inventory is determined by the results of an SLC mixing algorithm using a COG of 1% Cu and a SOG of 0.7% Cu.

The process used to apply the mixing algorithm includes:

- Creating a large array of stope shapes that includes the likely ore zone and waste zones (to represent dilution) above and adjacent to ore zones.
- Evaluate the stope shape array for tonnes and grade.
- Run the mixing algorithm using the selected COG and SOG.
- Generate footprint strings of potential mining areas.

- Modify footprint strings taking into account practical mining shapes and minimum mining widths.

Figure 16.3 illustrates an isometric view of the cave design.

**Figure 16.3 Isometric View Cave Design**

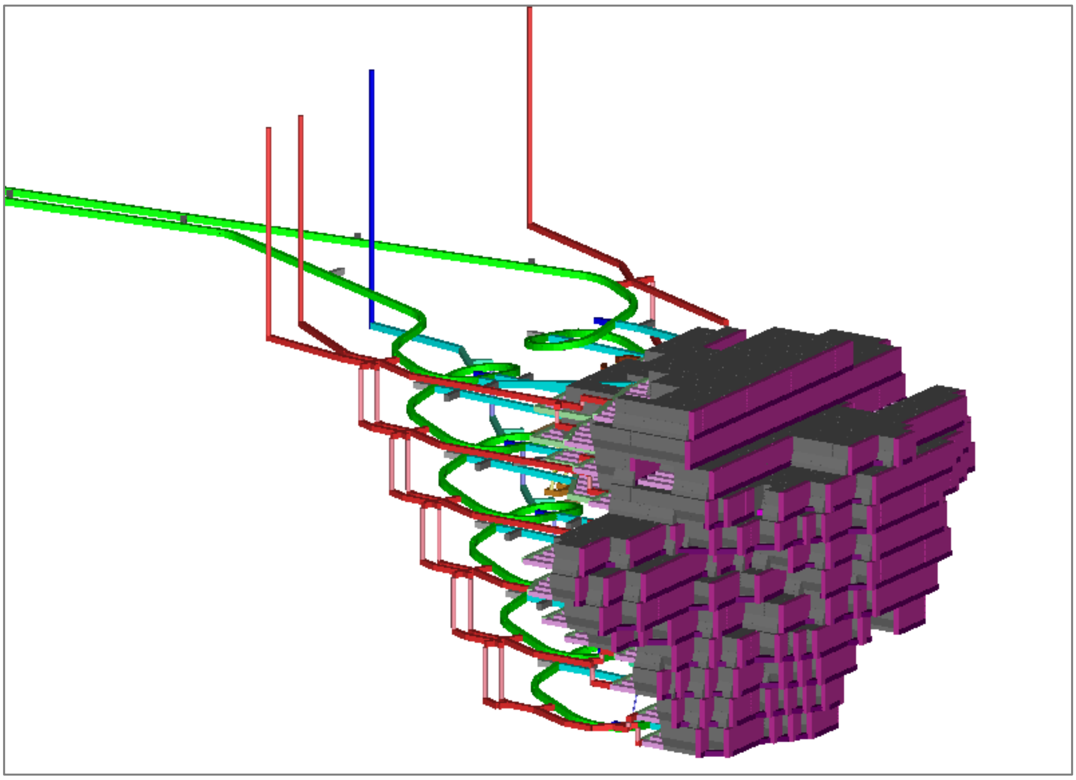
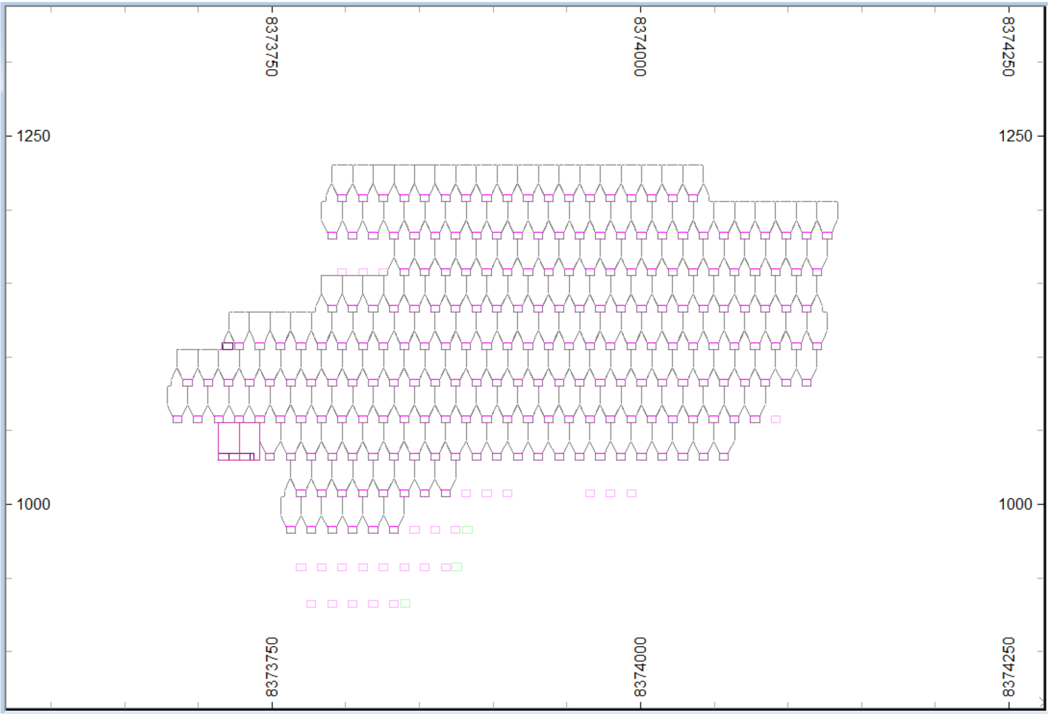


Figure 16.4 illustrates a cross-section of the stope design.

**Figure 16.4 Cross-section View Stope Design**




## 16.6 Underground Equipment

Underground mining is based on using mechanized mobile equipment.

AMC has made cost allowances for the mobile equipment including:

- Underground heavy mobile equipment, including the following:
  - Twin boom development jumbos.
  - Production drill rigs.
  - Underground loaders.
  - 55 t dump trucks.
  - Charge up machines.
  - A water truck.
  - Service vehicle (Integrated tool carrier).
  - A cable bolt machine.

  - Grader.

- Underground light vehicles, including the following:

  - A workshop forklift.

  - A surface bus.

  - Utilities and wagons.

  - Explosives transport utility.

  - 6 tonne light truck.

  - Mines rescue vehicle.

Table 16.6 lists equipment productivities and average utilized hours per period, applied in the schedule in the cost model.

**Table 16.6 Schedule Equipment Productivity and Average Utilisation Hours**

| Description | Unit | Maximum Productivity | Average Utilised Hours (Hours/month) |
|---|---|---|---|
| Jumbo drill | m/month | 200 | 150 |
| Production loader | t/day | 2,500 | 485 |
| Production drill | pdm/month | 6,000 | 250 |
| Truck | Mtkm/year | 1 | 400 |
| Charge up machine | – | NA | 250 |
| Water cart | – | NA | 150 |
| Service vehicle (ITC) | – | NA | 150 |
| Grader | – | NA | 150 |
| Cable bolter | – | NA | 150 |

Table 16.7 lists equipment rates and maximum equipment numbers applied in the schedule.

**Table 16.7 Schedule Equipment Productivity and Numbers[1]**

| Description | Unit | Maximum Productivity | Maximum Number |
|---|---|---|---|
| Jumbo drill | m/mo | 200 | 5 |
| Production loader | t/d | 2,500 | 4 |
| Production drill | pdm/mo | 6,000 | 4 |
| Raise drill | m/d | 2 | 5 |
| Backfill[1] | $m^3$/d | 1,000 | 1 |

**1 Truck numbers are not limited in the schedule, numbers are based on a monthly productivity of 1 Mtkm per annum per truck.**

## 16.7 Mine Infrastructure

For the OPFS, AMC made allowances for major mine infrastructure and facilities which include:

- Mine dewatering.
- Underground services.
- Electrical distribution.
- Compressed air.
- Refuelling and service bay.
- Underground offices and crib room.
- Refuge chambers.
- Emergency egress.
- Underground magazines.
- Mine communications.

***Mine Dewatering***

The underground primary dewatering system design is based on a capacity of ~30 L/s to 50 L/s for both the north and south declines indicating a total capacity of ~60 L/s to 100 L/s. The system is designed to accommodate average inflows (based on hydrogeological assessment undertaken for the study) from fault zones assuming the fault zones are dewatered to remove possibilities of bust inflows prior to the underground mining intersection of faults.

8 kW portable submersible pumps or similar are carried on-board development and production drill rigs for dewatering during drilling operations. Water accumulated during the drilling operations is pumped to sumps on the access levels. These access level sumps are equipped with 18 kW submersible pumps (or similar type and capacity). Water from these level sumps is transferred by drainage line to the closest pump station and then pumped to the surface.

***Underground Services***

For cost estimation purposes, the underground services such as service water, potable water, electrical power and de-watering are assumed to be reticulated via the decline and other infrastructure development. Vent rises and/or service holes between production levels would be used as required.

Service water requirement for mining is assumed to be 10 L/s. For the underground mine water supply reticulation, a provision for the supply of 5,900 m of HDPE piping and fittings through decline has been made. AMC has also allowed for one 48 kL polyurethane tank as water supply head tank.

***Electrical Distribution***

Power to the underground mine will be provided via a feeder from the surface substation to the decline, where power will be distributed to the surface fans and then feed underground substations where it can be stepped down for use underground.

During the initial development phase of the mine, the electrical supply will be powered from a substation located at the mine portal. Once completed, armoured high voltage cables will be suspended down the raise bored shafts to substations located in north and south declines.

The electrical distribution system will utilise 11 kV high voltage and 1 kV low voltage networks.

### 16.7.1 Compressed Air Reticulation

Mine compressed air will be required for production drilling, pneumatic tools, operation of any hand held drilling (potentially used secondary blasting), charging operations, shotcreting and miscellaneous service crew requirements.

Compressed air demand has been postulated by the following equation:

$$\text{Mine Air Demand} = \text{Annual Broken Tonnes} \times 8\ \text{kNm}^3$$

AMC has allowed two skid based compressors.

Compressed air for mining production is fed to and down the declines via DN110 PE100 PN16 pipes.

### 16.7.2 Underground Refuelling and Service Bay

Underground refuelling and service bay, designed to cater for drills, loaders and service vehicles comprises the following:

- Service bay with jib crane.
- Refuelling bay with mobile fuel stations.
- Oil sump and separator.
- Fire suppression system.

It is anticipated that the service bay will be used for minor maintenance only. Major repairs will be carried out in the surface workshop.

### 16.7.3 Underground Magazine

An allowance has been made for an underground magazine comprising:

- 3 x magazine bays.
- Monorail.

Allowance has been made for the fit-out of an underground explosive magazine. The fit-out assumes separate storage of ancillaries and explosives in suitable containers and shelving with appropriate fire precautions and magazine security.

### 16.7.4 Refuge Chambers

Self-contained and relocatable refuge chambers are accounted in the capital cost estimate.

AMC has made allowance for 20 person refuge chambers for use during the development phase of mine as escapeway systems are established.

AMC has also made allowance for 4 person portable refuge chambers for use in areas where a second means of egress cannot be provided.

### 16.7.5 Emergency Egress

The mine design includes twin declines connected at intervals. Consequently over the majority of the mines vertical extent there are two means of egress from the mine.

On the lower levels of the mine where a single decline progresses, escapeway raises and ladder ways are included for provision of a secondary means of egress. AMC has made allowance for the supply and installation of modular escapeway ladders, fully caged with rest landing spaced at 6 m intervals while inclined at 70º or steeper.

#### 16.7.6 Mine Communications

Allowance has been made for a VHF leaky feeder system. All mobile equipment will be equipped with radio sets. Key labour and supervision staff will be provided with handheld radio sets to provide communication on dedicated chat channels. Radios will also be installed in offices (such as the technical, emergency response, and first aid offices). Emergency response will have its own dedicated channel.

### 16.8 Ventilation

The mine is designed to be ventilated by a 'pull' or exhausting type ventilation system. That is, the primary ventilation fans will be located at the primary exhausts of the mine and develop sufficient pressure to provide ventilation to all workings from the intakes through to the exhaust system and to the surface. Air is drawn into the mine through the two declines and the single fresh air rise network.

The following further general criteria are also established:

- Air residence time will be kept as short as possible to minimise personnel exposure to dust, heat, diesel particulates and other contaminants.
- Each level will be developed such that an exhaust route is established prior to commencement of production on that level.
- Recirculation of air is entirely prohibited.
- Series ventilation will be kept to an absolute minimum and only if a suitable quantity of fresh air is introduced at the start of the series.
- The use of ventilation doors and in particular airlock doors in ramps will be avoided where possible.
- Regulators will be used to control and redistribute the quantity of flow in each split of air.

Air volume requirements are calculated to ensure safe operation of the mine producing at 3 Mtpa. The amount of air required is largely determined by the number and engine size of diesel equipment operating underground as well as ventilation of infrastructure; namely the ore passes. The air volume supplied must be able to dilute and remove dust and noxious gases as well as diesel particulate matter generated by the use of diesel equipment.

A contingency of 25% is applied to the final calculated air volume to account for potential leakage through the cave and transient system inefficiencies.

All main fan installations are located on surface and will be controlled with variable frequency drives to allow fluctuation in air volumes during the life of the mine as well as ensure effective dust and contaminant removal.

In regard to design of secondary systems (dead-end headings that are not part of the primary ventilation network), consideration will be given to leakage, expected maximum duct length, duct diameter and volume to be delivered to the face.

Noting the cumulative total airflow for each year, the highest required airflow is estimated to be 615 $m^3/s$.

The ventilation system for Kitumba was modelled using Ventsim Visual™ Advanced. This software provides for three dimensional visualisation of a network and uses a form of the Hardy-Cross method for the ventilation network calculations.

## 16.9 Personnel

AMC has estimated labour costs for Kitumba based on labour numbers and estimated labour costs. The majority of labour costs are driven by equipment numbers (equipment operators and maintenance personnel). Subsequently, AMC have added allowances for management, supervision, technical services and support.

Labour costs are based on an operation that will comprise a three panel, two shift, where there is:

- Continuous operation, 24 hours per day, 365 days per year.
- Two 12 hour shifts per day.

Labour numbers are based on the employment of both Zambian and expatriate labour, employing different rosters. Personnel based in Zambia are scheduled to work a 2 week on and 1 week off roster. Expatriate personnel are scheduled to work a 6 weeks on and 3 weeks off roster.

## 16.10 Mine Schedules

The mine design is scheduled in MineRP's EPS. Design tasks are scheduled based on applied priorities, task rates and, equipment productivities and levels.

AMC prepared the mine schedule taking into account the distribution of grade. The high grade domain (used in estimating the Mineral Resource) is located towards the south of the mine design. Consequently the mine schedule was prepared to retreat from south-to-north, to access higher grade areas as soon as possible.

Table 16.8 lists task rates applied in the schedule.

**Table 16.8 Schedule Activity Rates**

| Description | Unit | Maximum Rate |
|---|---|---|
| Infrastructure Development | m/month | 150 |
| Ore Development (XC and Slot Drive) | m/month | 100 |
| Production Drilling | pdm/month | 6,000 |
| Vertical Development | m/day | 2 |
| Slot Rise Development | m/day | 2 |
| Stope Production (column position 1 and 2) | t/d | 250 |
| Stope Production (column position 3 and down) | t/d | 500 |
| Backfill[1] | $m^3/d$ | 1,000 |
| Backfill preparation (walls and distribution)1 | days | 7 |
| Backfill curing[1] | Days | 28 |

**1 Backfill related rates are not applied in the SLC option, but are applied in the alternate mining method, as discussed in Section 6.11.**

Table 16.9 lists equipment rates and maximum equipment numbers applied in the schedule.

**Table 16.9 Schedule Equipment Productivity and Numbers[1]**

| Description | Unit | Maximum Productivity | Maximum Number |
|---|---|---|---|
| Jumbo drill | m/mo | 200 | 5 |
| Stope loader | t/d | 2,500 | 4 |
| Production drill | pdm/mo | 6,000 | 4 |
| Raise drill | m/d | 2 | 5 |
| Backfill[2] | $m^3/d$ | 1,000 | 1 |

**1 Truck numbers are not limited in the schedule, numbers are based on a monthly productivity of 1 Mtkm per annum per truck.**
**2 Backfill scheduled in alternate design option.**

The schedule has been prepared so that development, stope drilling and ore production can occur on the same level at the same time. Tasks are restricted so that only one jumbo, production drill and stope loader can be active on a level at a time. Tasks are prioritised by level, activity type, proportion of primary sulphide and copper grade.

Figure 16.5 shows annual ore production and copper grade.

**Figure 16.5 Annual Ore Production and Copper Grade**

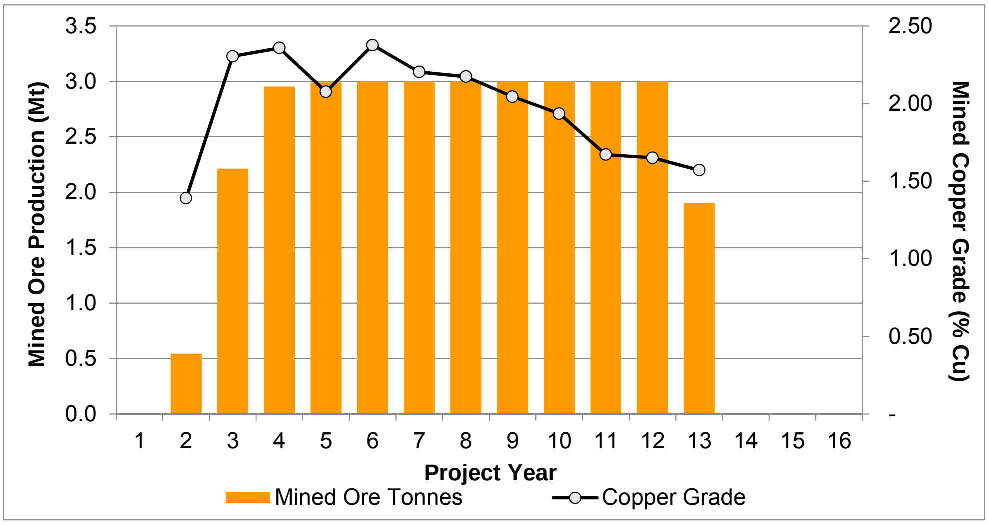
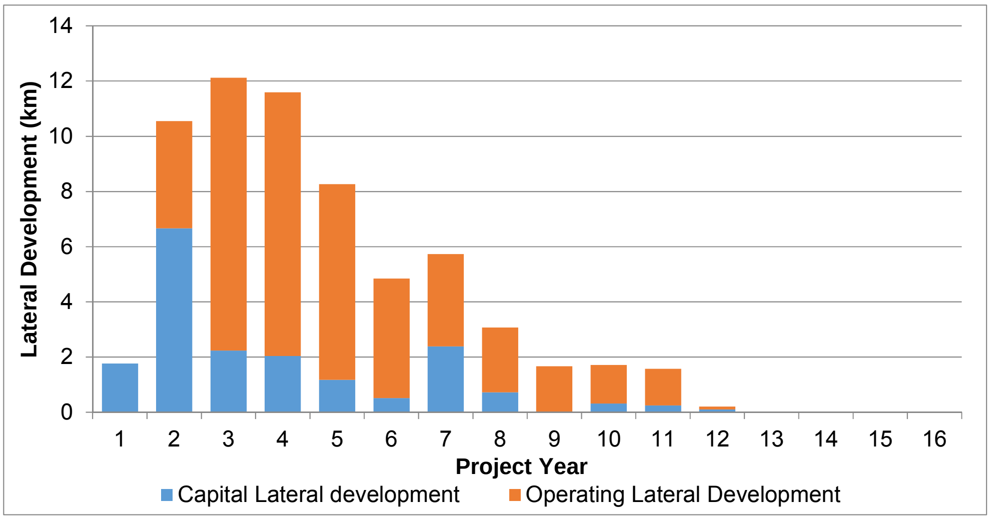



Figure 16.6 shows annual lateral development metres.

**Figure 16.6 Annual Lateral Development Metres**




## 16.11 Production Schedule

In addition to the mine schedule, AMC prepared a plant feed schedule for consideration by Lycopodium.

The plant feed schedule was prepared to:

- Take into account the feed ore throughput ramp up during plant commissioning.

- Blend ore to target a head grade (2.22% Cu) so that the mill is feed 3 Mtpa of ore and 66.7 ktpa of copper metal (60 ktpa recovered copper using a 90% metallurgical recovery).

- Prioritise processing of ore types, based on an AMC applied arbitrary blending classification. The arbitrary blending classifications comprise:

  - Type 1 – Primary sulphide tonnage greater than 1%, copper grade greater than 2.22% copper

  - Type 2 – Primary sulphide tonnage greater than 1%, copper grade less than 2.22% copper

  - Type 3 – Primary sulphide tonnage less than 1%, copper grade greater than 2.22% copper

  - Type 4 – Primary sulphide tonnage less than 1%, copper grade less than 2.22% copper.

- Type 1 and Type 2 ore types were prioritised in order to process as much primary sulphide material as possible to assist with acid and heat generation in the processing plant (shortfall to be made up by importation of concentrates and elemental sulphur), and to reduce residence time in stockpiles (to minimise the effects of oxidation).

- Minimise stockpile inventories.

The information provided by Lycopodium relating to the plant, and used to prepare the plant feed schedule includes:

- A 24 month plant construction period.

- A 9 month commissioning period following construction, where plant feed throughput commences at 100 ktpm and incrementally ramps up to 250 ktpm.

- Underground development is scheduled to commence 18 months before the first month of plant commissioning. Prior to the commencement of underground development AMC has scheduled a three month period of boxcut mining. Underground infrastructure development continues for 12 months before ore development begins, and initial undercut stoping commences four months after that. Ore production then gradually ramps up until it reaches 3 Mtpa, 33 months after the commencement of underground development.

- A mine life of 11 years.

- Figure 16.7, Figure 16.8, Figure 16.9, and Figure 16.10 illustrate the mined ore production, stockpiled inventory, ore tonnes processed and contained copper metal processed respectively, by month and by blending classification.

**Figure 16.7 Mined Ore Production by Blending Classification**

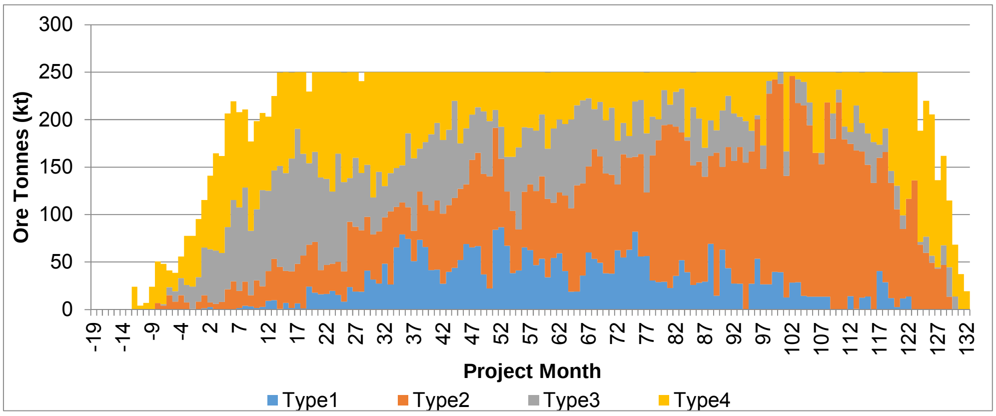



**Figure 16.8 Stockpiled Ore Inventory by Blending Classification**

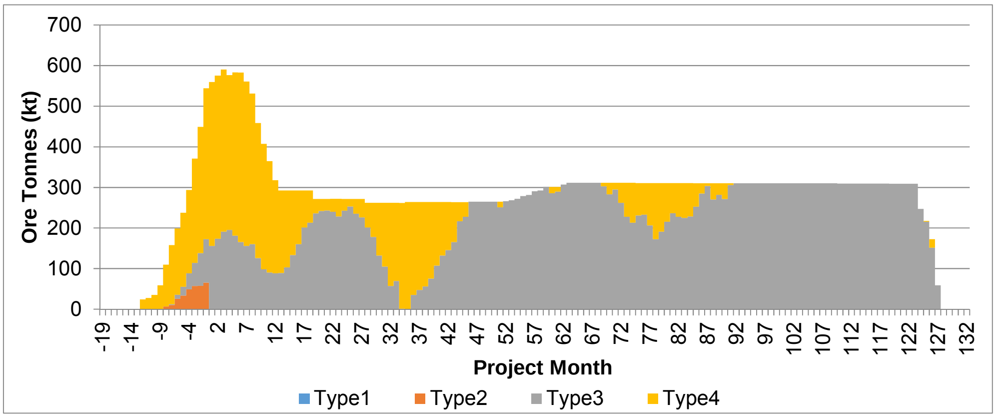



**Figure 16.9 Processed Ore Tonnes by Blending Classification**

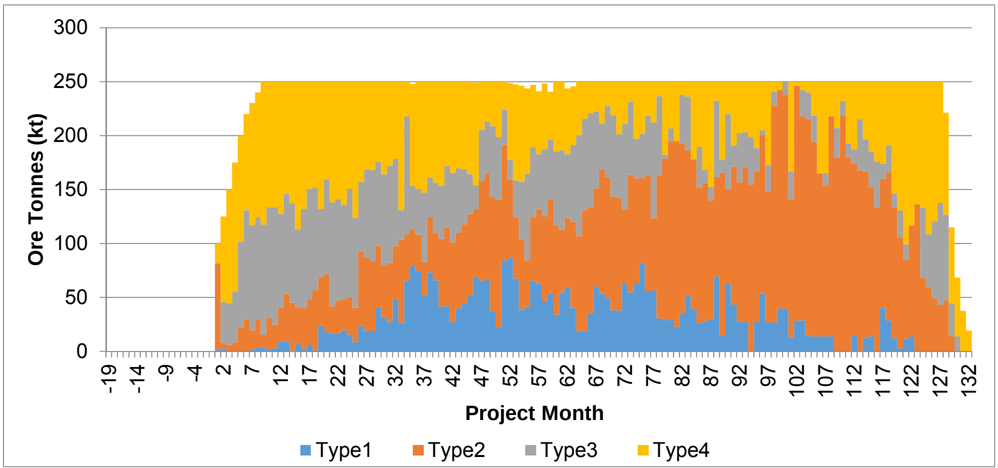
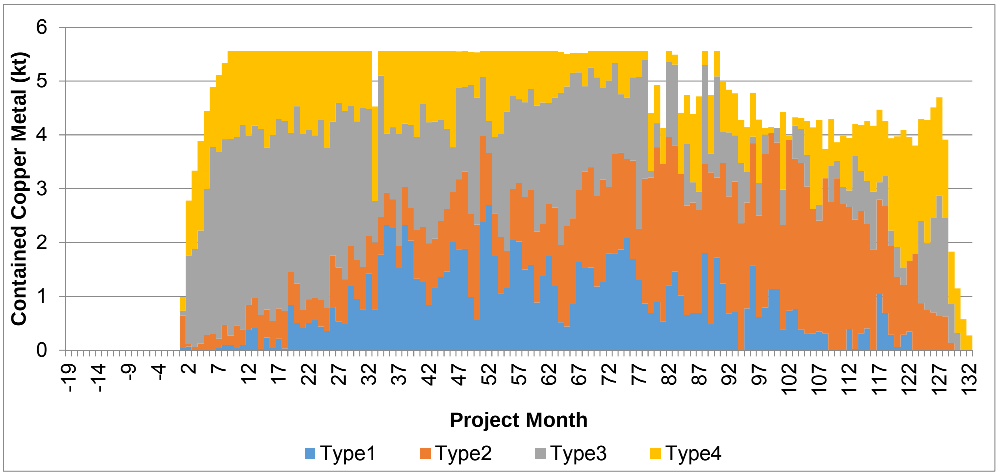


**Figure 16.10 Contained Copper Metal by Blending Classification**




## 16.12 Costs

AMC estimated mining capital and operating costs for Kitumba from first principles, based on the mine design and schedule.

AMC estimated mining costs to a pre-feasibility study level of accuracy (±25%). Cost are estimated from first principles using information from AMC's cost library, and information sourced from mining contractors operating in Zambia. AMC has not made allowance for a cost contingency.

Mining costs are based on the work being undertaken by an experienced underground contractor that will provide supervision, labour, equipment and consumables, while Blackthorn will provide mine management and technical services.

The mining cost estimate includes allowance for a 15% contractor's margin and an additional 5% allowance to allow for contractor's overheads costs. The contractor's margin and overhead cost is not applied to electrical power or diesel, as these consumables are to be provided by Blackthorn.

Costs estimates are in real 2014 United States Dollars (US$).

The underground mining costs are based on:

- Mine planning and scheduling for a project producing 3 Mtpa of ore.
- Budget quotations for mining equipment provided by suppliers.
- Budget quotations for mining consumables provided by suppliers.
- Australian and other expatriate labour cost are based on the 2014 Hays Salary Survey.
- Zambian labour costs based on a current contractor costs.

- An initially high proportion of expatriate labour, which is gradually reduced.
- Sustaining capital of 2% per year on infrastructure capital.
- Ground support options requiring the installation of splits sets and weld mesh.

Blackthorn provided AMC with rates for diesel ($1.45/L) and electricity (9.7c/kwh).

Underground development costs include allowances for bolting and meshing, and intersection cable bolting, as indicated by geotechnical analysis. Allowance is also made for shotcrete application equipment.

The mining cost estimate includes allowances for:

- Underground infrastructure, fixed plant and ancillary equipment.
- Mobile equipment.
- Labour.
- Consumables, diesel and electricity.
- Vertical development.
- Overheads and miscellaneous costs.
- Grade control drilling.

Capital costs estimated for Kitumba are summarised in Table 16.10.

**Table 16.10 Kitumba Mining Capital Costs**

| Item | Initial Cost ($M) | Sustaining Cost ($M) | Total Cost ($M) |
|---|---|---|---|
| Boxcut and portal | 5.8 | - | 5.8 |
| Heavy mobile equipment | 5.8 | 3.0 | 8.7 |
| Ancillary equipment | 3.8 | 0.2 | 4.0 |
| Primary ventilation fans | 1.7 | 0.1 | 1.9 |
| Infrastructure | 7.4 | 0.3 | 7.8 |
| Lateral development | 11.6 | 4.7 | 16.4 |
| Labour | 45.4 | 7.7 | 53.0 |
| Fuel | 2.0 | 1.2 | 3.1 |
| Electrical power | 1.0 | 0.4 | 1.4 |
| Vertical development | 18.7 | 5.1 | 23.9 |
| Grade control diamond drilling | 0.3 | 32.6 | 32.9 |
| **Total Capital Cost** | **103.5** | **58.0** | **161.9** |

The total mining capital cost per tonne of ore is estimated to be $5.10/t.

Operating costs estimated for Kitumba are summarised in Table 16.11.

**Table 16.11 Kitumba Mining Operating Costs**

| Item | Pre-Production ($M) | Production ($M) | Total Cost ($M) |
|---|---|---|---|
| Underground equipment | 12.8 | 240.2 | 253.0 |
| Labour | 25.2 | 378.9 | 304.0 |
| Fuel | 1.5 | 59.8 | 61.3 |
| Electrical power | 1.1 | 18.8 | 19.9 |
| Horizontal development | 8.2 | 63.6 | 71.7 |
| Long hole stoping | 3.3 | 85.3 | 88.6 |
| **Total Operating Cost** | **52.1** | **725.8** | **777.9** |

The total unit operating cost per tonne of ore is estimated to be $25.30/t (excluding accommodation and messing costs, which are included in Lycopodium's cost estimates).

Table 16.12 lists mining costs by project year.

**Table 16.12 Mining Costs by Project Year**

| Year | Year -2 ($k) | Year -1 ($k) | Year 1 ($k) | Year 2 ($k) | Year 3 ($k) | Year 4 ($k) | Year 5 ($k) | Year 6 ($k) | Year 7 ($k) | Year 8 ($k) | Year 9 ($k) | Year 10 ($k) | Year 11 ($k) | Total ($k) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Capital Costs** | | | | | | | | | | | | | | |
| Boxcut | 5,606 | - | - | - | - | - | - | - | - | - | - | - | - | 5,606 |
| Portal | 200 | - | - | - | - | - | - | - | - | - | - | - | - | 200 |
| Major Equipment | 2,828 | 2,946 | 906 | 645 | 296 | 126 | 779 | 171 | - | 8 | 7 | 23 | - | 8,735 |
| Ancillary equipment | 3,364 | 404 | 68 | 39 | 18 | 8 | 56 | 13 | - | 1 | 0 | 2 | - | 3,973 |
| Ventilation | 999 | 746 | 43 | 27 | 12 | 5 | 20 | 4 | - | 0 | 0 | 1 | - | 1,857 |
| Underground Infrastructure | 6,248 | 1,159 | 187 | 27 | 20 | 3 | 104 | 5 | - | 0 | 1 | 0 | - | 7,754 |
| Underground Horizontal Development | 5,129 | 6,489 | 1,628 | 1,056 | 486 | 159 | 1,151 | 229 | - | 9 | 6 | 23 | - | 16,366 |
| Labour | 25,742 | 19,608 | 3,185 | 1,345 | 612 | 273 | 1,817 | 407 | - | 21 | 15 | 57 | - | 53,082 |
| Underground Fuel | 1,071 | 898 | 290 | 231 | 107 | 53 | 380 | 82 | - | 4 | 3 | 12 | - | 3,130 |
| Underground Electrical Power | 305 | 663 | 137 | 86 | 39 | 16 | 98 | 22 | - | 1 | 1 | 3 | - | 1,370 |
| Underground Vertical Development | 1,021 | 17,716 | 690 | 278 | 919 | 67 | 1,400 | 708 | 225 | 250 | 134 | 451 | - | 23,859 |
| Underground Diamond Drilling | - | 318 | 2,420 | 3,190 | 3,142 | 3,169 | 3,069 | 3,109 | 3,116 | 3,082 | 3,081 | 3,142 | 2,054 | 32,893 |
| Sustaining Capital Infrastructure | - | - | 163 | 292 | 299 | 304 | 319 | 323 | 324 | 327 | 343 | 350 | - | 3,044 |
| Contingency | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| **Total Capital** | **52,513** | **50,948** | **9,717** | **7,216** | **5,949** | **4,182** | **9,193** | **5,072** | **3,665** | **3,703** | **3,591** | **4,064** | **2,054** | **161,867** |
| **Operating Costs** | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Underground Equipment | - | 12,849 | 24,307 | 25,978 | 25,246 | 25,020 | 22,174 | 21,630 | 21,475 | 21,265 | 21,536 | 17,549 | 13,949 | 252,977 |
| Labour | - | 25,165 | 37,812 | 25,612 | 25,581 | 25,414 | 23,052 | 23,964 | 24,134 | 24,040 | 23,894 | 22,988 | 22,374 | 304,030 |
| Underground Fuel | - | 1,479 | 4,128 | 5,361 | 5,592 | 6,113 | 5,864 | 5,986 | 5,917 | 5,656 | 5,571 | 5,382 | 4,221 | 61,271 |
| Underground Electrical Power | - | 1,086 | 1,990 | 2,056 | 1,977 | 1,881 | 1,543 | 1,585 | 1,584 | 1,580 | 1,556 | 1,538 | 1,528 | 19,904 |
| Underground Horizontal Development | - | 8,147 | 13,825 | 13,641 | 9,788 | 6,124 | 7,136 | 4,271 | 2,519 | 2,569 | 2,411 | 783 | 531 | 71,746 |
| Underground Long Hole Stoping | - | 3,327 | 13,003 | 11,156 | 11,855 | 11,064 | 7,668 | 7,387 | 7,088 | 6,854 | 4,900 | 2,542 | 1,751 | 88,595 |
| Contingency | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| **Total Operating** | **-** | **52,053** | **95,066** | **83,804** | **80,040** | **75,617** | **67,437** | **64,823** | **62,716** | **61,964** | **59,869** | **50,781** | **44,354** | **798,523** |
| **Total Costs** | **52,513** | **103,001** | **104,783** | **91,020** | **85,989** | **79,799** | **76,630** | **69,895** | **66,381** | **65,667** | **63,460** | **54,846** | **46,408** | **960,390** |

# 17.0 RECOVERY METHODS

## 17.1 Process Flowsheet Development

The flowsheet for the Kitumba Copper Project has been designed to process 3 Mtpa of ore to produce nominally 60,000 tpa of copper cathode, although peak copper output of 70,000 tpa may be achieved by increasing the plating current density from a design 310 A/m$^2$ up to 357 A/m$^2$.

An earlier flowsheet utilising flotation and atmospheric leaching, as adopted for the initial pre-feasibility study, was only able to achieve 76% copper recovery. Thus the main focus in developing the optimised flowsheet was to increase the copper recovery, while remaining cognisant of containing capital and operating cost, in an attempt to improve the overall project economics. The flowsheet as adopted for the OPFS, although more complex than its predecessor, is able to achieve >90% copper recovery at a significantly lower unit operating cost of production. Despite the added complexity, the individual unit processes adopted are well understood and have been selected based on successful operating practices on a number of commercial copper processing operations worldwide.

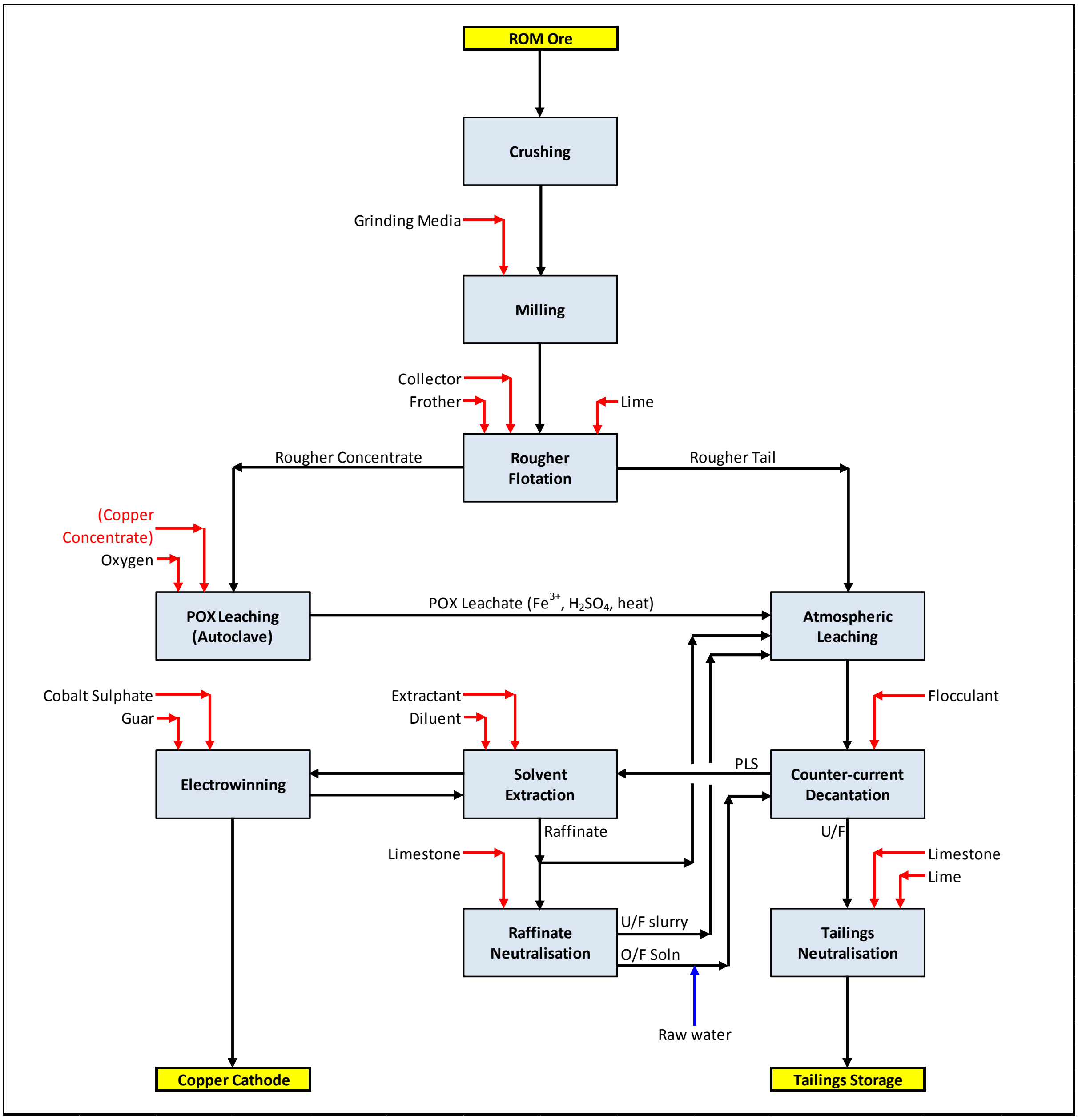
The processing facility comprises a comminution circuit followed by flotation to produce a sulphide rich concentrate. The flotation concentrate is then leached in a pressure oxidation (POX) autoclave while the flotation tail is combined with the POX leach discharge slurry in a series of atmospheric leach tanks to solubilise the residual copper. Sulphuric acid and ferric species generated in the POX circuit are used to affect copper dissolution, while the steam generated from the oxidation of sulphur is also used to heat the incoming atmospheric leach feed slurry to enhance leach kinetics.

The discharge slurry from the atmospheric leaching circuit is washed in a series of six counter-current decantation (CCD) thickeners to separate the copper rich pregnant leach solution (PLS) from the barren leach solids. The washed barren solids are neutralised and discarded to a tailings storage facility, while the PLS is fed to a solvent extraction circuit. Loaded strip solution is fed to a copper electrowinning tankhouse for subsequent generation of a copper cathode product. Residual soluble copper is precipitated from a bleed stream of solvent extraction raffinate solution and recycled, while the barren liquor provides the wash solution for the CCDs.

A series of batch flotation, leaching, settling, and neutralisation tests has been carried out between November 2013 and February 2014 and the results of these tests have been used to generate the process design criteria. The process design criteria have in turn been used to develop a series of mass and energy balances for the revised flowsheet, the outputs from which have been used to size mechanical equipment and to estimate reagent and utility requirements. Although the initial batch tests have provided a substantial insight into the behaviour of the leaching processes, a pilot campaign is planned which will provide a full set of criteria for the Definitive Feasibility Study (DFS). Consequently there remain a number of assumptions in the process design criteria for this optimised pre-feasibility study, but the design presented below is considered to provide a reasonable basis on which to assess the project.

The overall block flow diagram for the revised process plant flowsheet is shown in Figure 17.1.

**Figure 17.1 Process Plant – Block Flow Diagram**




The most significant change to the Kitumba process plant flowsheet from the previous 2013 PFS involves the ability to achieve greater overall copper recovery by the inclusion of a pressure oxidation leaching step. The production of a lower value flotation concentrate is no longer included in the flowsheet as all concentrate will now be leached through the POX autoclave. Although the revised flowsheet requires significant quantities of oxygen in the POX leaching circuit, reduction in the importation of large quantities of sulphuric acid, coupled with the improved overall copper recovery delivers a considerably more robust and viable outcome. In addition, the revised flowsheet is able to accommodate the highly variable ore types likely to be encountered from the Kitumba deposit.

The process plant for treatment of the Kitumba ore includes the following unit operations:

- Primary crushing and milling in a SABC circuit to achieve a nominal $P_{80}$ of 150 µm.
- Rougher flotation of the mill product to produce separate concentrate and tails streams.
- Pressure oxidation (POX) leaching of the flotation concentrate in an autoclave to produce sulphuric acid, ferric species, and heat.
- Mixing of the flotation tails with heated raffinate solution and POX leach discharge slurry.
- Leaching of this combined slurry in an atmospheric acid leach circuit to yield a copper rich pregnant leach solution (PLS).
- Separation of the PLS from the barren leach solids in a six stage counter-current decantation (CCD) circuit.
- Extraction of copper from the PLS using solvent extraction.
- Recovery of copper as copper cathode using electrowinning.
- Precipitation and subsequent recycle of copper hydroxide from a bleed stream of solvent extraction raffinate using milled limestone. Recycle of the neutralised raffinate as wash solution in the CCD circuit.
- Neutralisation of the final CCD stage underflow slurry, using limestone and lime, prior to disposal.

A preliminary plant layout drawing has been included in Appendix 2.

The process design criteria are summarised in Table 17.1.

**Table 17.1 Summary Design Criteria**

| Description | Unit | Design Value |
|---|---|---|
| Life of Mine | y | 11 |
| Ore Milled, dry | t/y | 3,000,000 |
| Plant Operation | days/y | 365 |
| Plant Operating Hours | h/y | 8,000 |
| Milling Circuit Configuration | - | SABC |
| Milling Circuit Specific Power | kWh/t | 21.4 |
| Milled Product Particle Size, $P_{80}$ | µm | 150 |
| Flotation Concentrate Mass Pull | % | 7.5 |
| Atmospheric Leach Feed Density | %w/w | 25 |
| Atmospheric Leach Temperature | °C | 60 |
| Atmospheric Leach Residence Time | minutes | 480 |
| Atmospheric Leach Copper Extraction | % | 80 |
| POX Leach Feed S Content | t/y | 37,000 |
| POX Leach Feed Density | %w/w | 20 |
| Oxygen Addition / t S | t/t | 2.5 |
| POX Leach Operating Temperature | °C | 220 |
| POX Leach Oxygen Partial Pressure | kPa | 800 |
| POX Leach Operating Pressure | kPag | 3,110 |
| POX Leach Residence Time | minutes | 60 |
| POX Leach Copper Extraction | % | 98 |
| Copper Production (nominal) | t/y | 60,000 (as cathode) |

The comminution and flotation sections of the process plant will treat 3 Mtpa at a design feed rate of 375 t/h, and with an average 7.5% mass pull to concentrate.

A key component for the successful operation of the process flowsheet is the correct regulation of primary sulphide material reporting to the POX leaching circuit, in order to guarantee auto-thermal autoclave operation while generating sufficient reagent (sulphuric acid and ferric ions) and heat for the atmospheric leaching circuit. Consequently, a facility to import an additional source of sulphur (either as pyrite, chalcopyrite, or elemental sulphur) has been included in the design to supplement the POX leach feed if the flotation concentrate sulphur content is low. The Mine Schedule indicates that primary sulphide mineral deficiency will occur mainly within the first four years of production.

Conversely, towards the end of the mine life there will be more than sufficient primary sulphide material to sustain leaching operations. This will result in the need to oxidise a greater mass of sulphur, requiring a larger autoclave and, by association, a larger oxygen plant. The sizing of the autoclave, oxygen plant, and associated POX leaching equipment has taken due cognisance of the variable concentrate feed composition.

Reagent handling, mixing, and distribution systems will depend on the usage volume and delivery method:

- Liquid reagents will be dosed directly from the delivery isotainers where possible.

- Powdered reagents, such as collector and flocculant, will be delivered in bulk bags and made up to solution in dedicated mixing facilities on site.

- Limestone will be a large volume consumable and will be delivered in bulk, pre-crushed to 20 mm, and milled on site to form a slurry suitable for distribution.

- Quicklime will be delivered either in bulk bags or in bulk tankers, and slaked in a dedicated slaking facility on site.

- Oxygen will be provided as a high pressure gas on an 'over the fence' contract basis.

## 17.2 Process Design

The design basis for the process plant has been developed from a series of bench scale tests, with appropriate scale-up factors applied to testwork results.

Standard chemical relationships have been used in the development of the Metsim mass and energy balance and these have been ratified where applicable by the batch testwork results.

Variations in the feed ore will affect the mass flow through the different parts of the circuit. Provision of surge capacity and blending of imported concentrate and/or sulphur ahead of the POX leach autoclave allows some smoothing of feed grades to minimise the over-design requirements for individual sections. Nominal design flowrates for process streams have been taken from the mass balance for incorporation into the Process Design Criteria. Reagent consumption rates have been drawn from a combination of testwork results and experience, noting that there is potential for further optimisation.

The processing plant facility has been designed for 24 hour per day operation and an 11 year mine life. The design of the process plant, together with all services and utilities, reflects:

- A control philosophy that encompasses a level of instrumentation and control sufficient to maintain steady operation and optimum plant throughput.

- Sophisticated monitoring, alarming and diagnostics to facilitate troubleshooting.

- Capture and containment of spillage for return to the process circuit via sump pumps.

- Capture and scrubbing of all off-gas and vent streams before discharging the cleaned off-gas to atmosphere.

## 17.3 Engineering Design

In general, design capacity for equipment has been taken from the nominal process design flow rate with a design factor added to accommodate instantaneous surges.

In most instances, materials of construction for mechanical equipment have been nominated, based on Lycopodium experience, industry norms, and vendor recommendations. It is acknowledged that potential exists to use alternative materials to those currently specified – this will be evaluated during the next study phase.

In accordance with best engineering practices, the POX leach autoclave has been specified as carbon steel, protected by a polymer membrane, three layers of refractory brick lining, potassium silicate mortar and titanium linings and inserts on all nozzles. The highly corrosive atmosphere within the autoclave precludes the use of internal autoclave metallic components other than titanium. The use, however, of a fully titanium clad autoclave as a viable alternative to a brick-lined unit has been discounted due to the inherent danger of titanium fires in an oxygen-rich environment.

It is preferable to install the autoclave lining system in a controlled environment on the vendor's premises, and thereafter to transport the lined autoclave to site. However, this approach is subject to a route survey and the outcome of possible transport restrictions, and further investigation is required.

The engineering design takes cognisance of the conditions under which equipment and personnel will be required to operate by way of:

- Ease of maintenance, given the high monetary cost associated with downtime on a facility of this size. Each facility, including piperacks, will be accessible using a mobile crane. In some instances, permanent cranes and hoists will be provided to assist with routine maintenance and operational activities.

- The copper solvent extraction facility will be located separately to the main plant to comply with fire prevention codes.

- Dust and off-gas collection and scrubbing facilities will be included in the design to minimise emissions and to ensure compliance with air quality regulations.

# 18.0 PROJECT INFRASTRUCTURE

## 18.1 Water Supply and Management

### 18.1.1 AGES Study

Africa Geo-Environmental Services Gauteng (Pty) Ltd (AGES) conducted a groundwater and environmental water management specialist study (refer Section 3.0).

The following summarises the findings of this investigation relevant to the OPFS.

The study area is situated in the Kafue sub-catchment of the greater Zambezi River Basin. The Kafue River is situated approximately 30 km to the north of the study area and forms part of the main drainage systems within the region. Wetlands / marshes within the study area are mostly surface water supported.

The Kafue River has a perennial flow and as such is a potential source of water for the project.

In August 2012, a regional hydrocensus survey was conducted within the study area to establish a hydrogeological and hydrochemical baseline.

In February 2013, an updated regional hydrocensus survey was conducted as part of the water supply study spreading out the coverage area to a 20 km buffer zone around the project area, and updating and extending existing data to form a comprehensive hydrocensus data base for evaluation and interpretation purposes.

The initial hydrocensus represented field data for the dry season, with the updated hydrocensus collecting field data for the wet, rainy season.

Water levels were measured and groundwater samples taken with relevant site specific data recorded.

The site is underlain by a shallow and a deep aquifer system. The shallow aquifer consists of ferricrete and alluvium material and occurs from surface to depths of 10 mbgl (metres below ground level). The deeper, intermediate aquifer is formed by weathered / fractured bedrock and occurs from 40 mbgl to 200 mbgl depth. The deep aquifer beyond 200 mbgl consists of solid and fractured bedrock at varying intersection depths.

Twenty-two of the surveyed hydrocensus sites were selected for chemical analyses and the development of a water quality baseline database. The sample points were selected to represent the spatial extent of the site and incorporate all water application areas (villages, mining area and agricultural areas). Of the sites selected, 17 are groundwater samples, four are surface water samples and one sample is representative of rain water.

The water quality of both the groundwater and surface water is generally of good quality with only four samples slightly exceeding normal potable standards.

The regional aquifer can be classified as a Sole Source Aquifer assuming that more than 50% of the groundwater is utilised for domestic and/or livestock purposes and no alternative water resource is available if this aquifer is impacted on.

### 18.1.2 AGES Groundwater Modelling

The simulated Life-Of-Mine (LOM) radius of influence for the underground mine extends to approximately 1,050 m south west of the mining operations and 800 m north east. The simulated inflows of mine dewatering commences within Year 1 and increases to a maximum of 3,283 $m^3/d$ (38 L/s) to 4,150 $m^3/d$ (48 L/s).

Modelling results in a sulphate contamination plume originating from the mine waste and residue facilities, extending to the north-west and develop along local fault zones acting as preferential path lines. The contamination plume does not extend more than 1 km down gradient of the mine waste and residue facility. It should be noted that the tailings storage facility (TSF) design for the OPFS incorporates a soil liner to minimise seepage losses from the facility.

For the base case underground scenario, the underground workings will take approximately 10 - 20 years to flood and the decline to the mine will take in excess of 100 years to flood to equilibrium level of 1,275 mamsl, which is approximately 5 m below the initial groundwater level. No decanting would take place as the water level elevation in the mine void would be much lower than the decline shaft elevation.

Most important impacts envisaged for the proposed mining operation include:

- Mine dewatering during the LOM could impact negatively on the local groundwater quantity. As there are no existing neighbouring groundwater users within the radius of influence of dewatering, the impact is considered as moderate.

- Contamination of groundwater sources due to seepage from TSF and water storage facilities as well as seepage from point sources and contaminated storm water run-off. The impact is considered as moderate.

### 18.1.3 RPS Groundwater Modelling

AMC commissioned RPS to assess the effect that groundwater could have on the operation of the underground mine (refer Section 3.0).

The RPS report estimates water inflows to the mine will peak at 1,814 $m^3/d$ (21 L/s) in Year 2 and will decline progressively to 1,382 $m^3/d$ (16 L/s) in Year 5 and 1,037 $m^3/d$ (12 L/s) in Year 16 (the calculations were based on an early 16 year mine plan).

The peak RPS inflow prediction is thus only 50 percent of that predicted by AGES and occurs early in the mine life rather than progressively increasing throughout. The discrepancy between the predictions was referred to the consultants concerned with no definitive conclusion other than that, given the level of information available, either prediction could be correct or the actual flows could be between the two ranges predicted.

Additional work is planned during the DFS to obtain additional data for mine water inflow modelling to enable a more definitive prediction to be made. For the purposes of the OPFS KP used the lower RPS predictions when undertaking the life of mine site water balance. This is a conservative approach as it maximises the estimated shortfall of water required for the process plant.

### 18.1.4 Site Water Balance

A site water balance model was developed by KP and is reported in full in their TSF Optimisation Study Report appended to the OPFS report.

The current site water balance is based on construction of a single water storage facility of approximately 2 $Mm^3$ to contain water from the underground dewatering as well as runoff from the external catchment and waste dump area. This will be supplemented by water from a borefield drawing from a local aquifer.

The water storage facility will be constructed early as underground dewatering will start as soon as decline development commences. Early completion will enable two wet seasons of surface runoff to be harvested prior to plant commissioning. The water storage facility will be constructed with a low permeability liner layer to minimise seepage losses.

Although regional work undertaken by AGES indicates that significant underground water resources exist, the aquifers are yet to be tested to prove they are adequate to meet the projects annual water make-up demand of up to 900,000 $m^3$ (29 L/s) under average rainfall conditions and 1.5 $Mm^3$ (48 L/s) under 1 in 100 dry rainfall conditions. Additional work on groundwater resources in the area is planned for the DFS however, in the event that no suitable source can be found, the fall-back position is the piping of water from the Kafue River, a distance of approximately 30 km.

Pumping water from the Kafue River will have a capital cost impact of some $20 million over and above the cost to develop a local borefield (REDE Bulk Water Value Engineering Report, refer Section 3.0).

Evaporation data used for the site water balance is based on data obtained from literature research.

Various climatic conditions such as average, 1 in 250 wet and 1 in 100 dry were used to quantify the plant make-up water requirements and for the TSF embankment design.

No data was available to carry out a reliable estimation of PMP (Probable Maximum Precipitation). The PMP used for the TSF spillway design has been based on data obtained from another site in-country (Lumwana). The adopted PMP value is 251 mm in 24 hours based on a 1:10,000 year rainfall event. A PMP based on local data will be estimated during the next stage of project studies.

In order to assess the runoff response of the site catchments a runoff estimation and flood routing model was constructed using 'HEC-HMS' software.

Due to the very high rainfall intensity associated with this hydrograph, it is deemed a conservative approach. On progressing to detailed design stage, KP will derive a specific hydrograph for the region and conduct a more detailed runoff routing model to refine the calculations.

#### 18.1.5 Monitoring, Mitigation and Closure

Final designs will be developed to minimise the impact of the project on both surface waters (which flow to the Kafue River) and ground water.

During construction, sediment control dams will be established to capture silt mobilised from cleared areas with water monitoring points established both up and downstream of the development areas. These will be maintained as necessary during operation.

Sediment control on the site will comprise a series of collection ponds and diversion channels and toe drains. Two sediment control ponds will be located at the TSF, taking the run-off from the TSF downstream embankment which is run through the toe drain. The sediment ponds will be cleaned out periodically and the water will be collected and pumped back to the process plant if considered contaminated, or realised into the environment if considered clean.

A similar configuration is envisaged for the waste rock dump, with the run-off from the waste dump being pumped to the water storage dam.

Uncontaminated stormwater and runoff from catchments upstream of the project will, where they are not harvested for use, be diverted around disturbed areas to prevent contamination. Throughout the mine life contaminated water, either from mine dewatering or stormwater runoff from the process plant and other areas, will be captured and stored for use as process make up water.

All supernatant water from the TSF will be returned to the process plant.

Monitoring and seepage capturing boreholes will be drilled at specific positions and aquifer levels and water quality monitored. Wetlands within the site boundary will be monitored as part of a comprehensive environmental monitoring and management program.

The final site closure plan will detail how surface water will be managed post closure and will address any potential legacy issues. All infrastructure including pipelines, not required after closure will be dismantled and removed from site. Roads will be rehabilitated to original land use and vegetation. Where required, surfaces will be re-shaped to prevent excessive erosion and divert stormwater to existing water courses.

The geochemical modelling indicated that although there is a potential for the mining operations to produce acid, the mineral processing will result in significant changes to the geochemistry of the ore as it will be limed and neutralised as part of the metallurgical process prior to disposal. Small quantities of uranium also occur in the deposit, but with the base concentration at 8-30 ppm this falls below the regulatory limits of 80 ppm and thus is within exemption criteria.

## 18.2 Mine Waste

The estimated total waste rock generated for the project is 2 Mt comprising an approximate 50:50 split of material from the boxcut development and underground development waste including the decline. A waste rock dump has been located adjacent to the mine boxcut to minimise haulage distances. The dump will be designed with sediment and seepage collection structures and contaminated runoff will be pumped to the water storage dam for subsequent use as plant process water.

It is anticipated, however, that much of the mine waste generated will be used as a source of construction material for the tailings storage facility.

Waste remaining on surface at the end of the mine life will be contoured and rehabilitated.

## 18.3 Tailings Storage Facility

KP was commissioned by Blackthorn at the pre-feasibility study stage to conduct a study of the tailings which will be produced by the development of the Kitumba Project and provide a design for a suitable tailings storage facility (TSF).

Following a high level desk review of the scoping study, a site visit was conducted by a KP Engineer in January 2013 in order to assess the key geological and geotechnical features of the proposed development.

A comprehensive TSF Options Study, considering several options from unthickened tailings to paste tailings and partial trucking of tailings, was undertaken as part of the pre-feasibility study. The options study considered climatological conditions, TSF configuration and tailings properties as well as cost, operational and risk elements and, as a result, a thickened tailings option was adopted.

The TSF configuration selected is a side valley storage facility and will be located at the north western corner of the project site. The main considerations for the TSF location were:

- The footprint of the underground workings to make sure no infrastructure would be placed over the top or within the zone of influence with regard to mining induced subsidence
- The location of the access portal to the decline and resulting process plant location to minimise tailings pumping requirements
- The local topography to minimise the rate of rise so upstream rises using tailings could be considered.

The pre-feasibility study TSF design was reviewed in light of the potential change in tailings properties as a result of the process flowsheet adopted for the OPFS. The central concept was considered to be still appropriate but changes were made with respect to the basin design, notably the addition of a low permeability soil liner, as a result of the changes to the geochemical characteristics of the tailings.

### 18.3.1 Tailings Geochemistry

A geochemical assessment was conducted on a sample of tailings produced from the batch metallurgical testwork.

The tailings were found to have a low total sulphur content of 0.3% with the majority of the sulphur present as sulphate. As a result the maximum potential acidity from the sulphite is 1.1 kg sulphuric acid / tonne, which is low.

The acid neutralising capacity (ANC) of the sample was determined along with the carbonate content. The results show that although the sample has a high carbonate content not all of it provides acid neutralising capacity, indicating the presence of some iron or magnesium carbonates.

A net acid generation (NAG) test indicated that the tailings do produce a small amount of acid under extreme oxidising conditions resulting in the pH of the sample dropping to below 4.5. However, the main conclusion is the tailings can be considered as Potentially Acid Forming – Low Capacity.

The supernatant water quality was assessed to examine the solubility of various elements. The results give an indication of the water quality likely within the pond during operation but cannot be used to predict long term seepage quality from the facility.

Several parameters including fluorine, molybdenum and selenium exceed the guidelines for release from mining operations and livestock drinking water. In addition TDS, antimony, arsenic, mercury and sulphate exceed drinking water guidelines.

Based on the fact that several elements in the supernatant water are predicted to exceed the drinking water guidelines by a factor of 10 but less than 100 the TSF design will require the inclusion of a single layer geo-synthetic or soil liner. This liner system will also be suitable for containment of the Potentially Acid Forming – Low Capacity tailings.

Upon closure the TSF will require a simple waste rock cover and revegetation to isolate the tailings from the environment.

### 18.3.2 Tailings Storage Facility Design

The principal objectives of the design of the TSF for the Kitumba Project are as follows:

- To produce a design which complies with relevant local, national and international regulations, guideline and standards using best available technology.
- Permanent, secure and total containment of all tailings within an engineered disposal facility.
- Achieve a high density, consolidated tailings mass by employing controlled sub-aerial deposition of tailings.

- Control, collect and remove free draining liquids from the tailings during operation for recycling as process water to the maximum possible extent.
- Maintain excess storage capacity within the tailings storage facility to contain the design storm allowance.
- Provide an engineered structure to control discharges from the storage due to extreme events greater than the design storm allowance.
- To reduce seepage from the facility during operation and on closure.
- Cost-efficient utilisation of available material for embankment construction.
- Allow for rapid, effective and stable rehabilitation.
- Provide monitoring features for all aspects of the facility and associated works to ensure adopted performance standard can be measured and achieved.

The facility is designed to comply with internationally acceptable design limits which have been based on the ANCOLD guidelines.

A tailings sample from the batch metallurgical testwork was used to carry out the tailings physical testing.

The testing indicated that the rate of supernatant release for this sample was fast, with the majority of water released within a few hours. At a target TSF feed solids of 55 to 60%, the expected water release would be around 27 – 37% of the water in slurry, not accounting for rainfall and evaporation but considering the loss of water to re-saturate lower tailings layers. Due to the high permeability, underdrainage would be expected to be around 10 – 20%.

With regard to the tailings density, the test results indicated that the tailings achieved medium to high densities (void ratio of around 1.1) from undrained settling. There was an improvement due to air drying with the void ratio reducing to around 0.7. It has been observed over a number of years that densities achieved in the field are generally lower than those obtained in the laboratory.

A typical achievable density range of between 1.50 t/m$^3$ for the early phases of the project to 1.65 t/m$^3$ in the later stages is expected.

The embankment has been designed as a zoned embankment with an upstream low permeability zone (Zone A) to reduce seepage from the tailings through the embankment and a downstream structural and erosion protection zone (Zone C). The first three stages are downstream construction as the rate of rise of the tailings is too high to allow sufficient consolidation and drying of the beaches to permit upstream construction. Stages 4 to 11 will be constructed as upstream raises.

The construction materials for the first three stages will be won from the basin area of the final TSF.

As the tailings from the adopted process flowsheet present a higher geochemical risk than those tested during the pre-feasibility study the use of tailings as a construction material for the embankment will not be viable and locally won borrow material will be used in the construction of all the upstream raises.

The tailings embankment has been designed as a lined facility. Based on the site inspection previously conducted it has been assumed that suitable soil will be available either within the TSF basin or from local borrow areas.

Deposition of tailings into the facility will occur from a line of spigots installed in a pipeline located on the embankment to form a tailings beach upstream of the embankment and push the ponded water within the facility away from the embankment. Additional deposition will occur from spigots installed within the pipeline upslope along the eastern edge of the TSF. The deposition into the facility will be rotated between deposition points to allow the tailings to form beaches between deposition cycles which will have the opportunity to desiccate and consolidate.

A detailed deposition strategy will be included in the next stage of project development.

Ponded water from the slurry will be decanted using electric pumps. The pumps will be placed inside a concrete decant tower, comprising 1,800 mm diameter concrete rings, placed inside a coarse rockfill causeway embankment. Several decant towers will be required through the life of the facility, each approximately 6 m in height. A causeway or access to the decant system will be provided along which the decant pipeline and power supply for the pumps will be laid.

A TSF surface water diversion channel prevents the clean surface water from entering the downslope TSF impoundment during normal rainfall events. Similar diversion channels will prevent clean storm water run-off from entering other contaminated parts of the site such as the ROM pad and plant site.

A cut-off trench is intended to reduce shallow seepage under the TSF embankment. The cut-off trench will be constructed continuously below the low permeability zone (Zone A) of the embankment and backfilled with low permeability fill (Zone A).

Two types of seepage control are adopted in the design of the TSF. A toe drain is proposed along the upstream toe of the embankment and underdrainage aimed at intersecting vertical seepage flow from the supernatant pond. The key benefits of the underdrainage are:

- Reduction of seepage out of the basin by collecting seepage flows within the facility

- Drainage of the tailings mass, thus increasing the density of the tailings and providing a more efficient facility in terms of storage

- Increase in the strength of the tailings mass (i.e. by consolidation) immediately adjacent to the embankment.

Water from the underdrainage system will be collected in a sump positioned at the low point of the basin. During operation this water will be pumped back to the facility but on closure, when the seepage flow reduces with reduced water inputs, the system will be decommissioned.

The TSF is sited as a side valley embankment and as such is connected to a relatively large catchment area. In line with international guidelines, emergency spillways are required to prevent overtopping of the embankment for each stage.

The spillway will outfall into a discharge channel, which in turn reports to a natural watercourse downstream. The spillway size has been designed to allow safe release of flow, or PMF, as a result of a Probable Maximum Precipitation event (PMP) and is the same for each stage.

### 18.3.3 Tailings Storage Facility Closure Considerations

The TSF embankment will be designed to ensure that it is inherently stable on closure and that the downstream face of the embankment will not present an increased risk to downstream stakeholders.

Detailed stability modelling will be conducted during the final design stages and at regular intervals throughout the operation of the facility to ensure that the final profile can meet the required factors of safety given by a range of internationally accepted standards and guidelines, including ANCOLD and ICOLD.

Closure of the TSF comprises construction of a cover to prevent dust and allow vegetation to establish over time. The type of cover in terms of material will be specified following detailed studies and depends on elements such as available material, vegetation type(s) and final TSF surface.

It is likely that progressive rehabilitation will be adopted which can commence towards the final life of the facility once or when embankments have reached final height and the final tailing surface has been reached for at least some of the TSF.

When considering the TSF only, a permanent spillway at the upstream edge of the TSF will allow storm water falling onto the facility to drain and will prevent overtopping of the pool over the embankments. On the upstream side of the TSF, a cut-off drain and bund will be constructed to divert stormwater towards the spillway side of the TSF. The size and material used in the cut-of drain and bund will be estimated following a more detailed assessment of the preferred design storm event. The spillway is likely to be extended and will outfall in one or more drainage channels which will either be allowed to sheet flow out to minimise erosion or be directed towards an existing water course.

Stormwater that falls onto the embankment will be collected through drains that are constructed on intermediate benches. The collected stormwater will be diverted towards the same side of the TSF as the spillway and will be diverted towards or into the same system. The drains are likely to be constructed using a rock fill or riprap type material to prevent erosion of the drain and benches.

## 18.4 Site Access

The site is accessed via 52 km of dirt road from Mumbwa (route D181). The first 48 km is a public road running north-west to the Lubunga pontoon crossing the Kafue River, while the last 4 km is a private road on the mining lease. The road will require upgrading to the standard required for construction traffic and ultimately operations traffic including concentrate delivery and copper cathode shipment by truck from the site.

The topography of the existing road is generally flat or gently undulating. There are no significant cut and fill constructions, river or creek crossings, or steep rises, falls, or winding sections. There are several low sections that will require raising over culverts for flood protection, but in general upgrading of the road for construction traffic and operations will only require relatively straightforward widening, grading, surfacing and low point elevation.

Unrelated to the development of the Kitumba project, the Zambian government has announced its intention to upgrade the public portion of this road, and the basis for the study assumes that this will be done at no cost to the project and in time for construction traffic. Interim upgrading of the road for construction traffic may be required if the D181 upgrade is delayed.

There are no plans to provide an airstrip at the site but allowance has been made for a helipad for medivac / emergency use.

## 18.5 Power Supply and Distribution

Blackthorn is in discussions with ZESCO for the provision of power from the grid to the project. ZESCO has confirmed that the most suitable supply option would be to spur off the proposed North West 330 kV double circuit power line connecting Mumbwa substation and Kalumbila substation. The distance from the proposed line to the site is approximately 3 km.

The preliminary scope includes:

- Establish a 330 kV T-off facility on the Mumbwa - Kalumbila line.
- Construct a double circuit 330 kV line from T-off point to the mine substation.
- Install a 330 / 33 kV, 2x60 MVA step down substation adjacent to the process plant.
- Install a new protection and SCADA system.

The substation will have a 33 kV panel room, battery room, protection and SCADA room, a control room with HMI for interface with other substations.

The new 330 kV transmission line works will include:

- Establishing and securing a Right of Way (ROW) access, including any necessary environmental approvals and land access.

- Construction using a lattice type structure or monopoles, as acceptable to ZESCO.
- Installation of a new line between the T-off point and the mine substation, a distance of about 3 km.
- Provision of two aerial earth wires, optical ground wire (OPGW) and one standard earth wire.

A dedicated step down substation will be established near the process plant for power distribution to the plant, mine and accommodation camp.

The main surface distribution voltages will be 33 kV, 11 kV and 415 V with underground power distribution at 1,000 V. The peak average site power demand is estimated at 45 MW with a more usual demand of 42 MW.

Emergency power supply for the essential loads of the process plant and mine during any grid power outage will be supplied by 3 x 1 MW diesel generators (i.e. two units on duty and one unit on standby). Transfer of power supply between grid supply and diesel generator or vice versa will be done manually as no automatic change-over facility will be provided.

## 18.6 Potable Water and Sewage

Raw water will be supplied from bores to a water treatment plant located at the plant site. The water treatment facility will include sand filtration, micro filtration, ultra-violet sterilisation and chlorination. Potable water will be stored in the plant potable water tank and will be reticulated to the plant building, site ablutions, safety showers and other potable water outlets. Additional ultra-violet sterilisation units will be installed on outgoing potable water distribution headers.

Potable water for the accommodation camp will be pumped from the water treatment plant at the plant site to a water storage tank at the camp. The tank will provide reserve storage in the event of supply line disruptions. Water will be delivered into the village reticulation system using a constant pressure variable flow pump system. The pump skid will include a UV disinfection unit to provide additional security against contamination.

Effluent from all water fixtures in the plant and accommodation camp will drain to a gravity sewerage system. The gravity sewerage system for each area will drain to a sewer pump station from where it will discharge via a pressure main to a packaged sewage treatment plant system.

Treated effluent will be discharged into the tails hopper and sludge will be suitable for direct landfill burial in unlined pits.

## 18.7 Surface Buildings and Facilities

Surface buildings and support facilities will generally be industrial type structures. Most will be constructed of a concrete slab on ground with structural steel frame and metal cladding. Offices and amenity buildings will generally be blockwork buildings. Facilities to be provided are listed below:

- gate house and security building
- general administration building
- clinic and emergency response facilities
- plant mess
- shift change room and ablutions (catering for surface and underground employees)
- laboratory
- plant control room and shift office
- central workshop / warehouse complex
- reagent stores.

Services will be provided to the mine and associated surface items such as ventilation fans as well as mining contractor facilities such as a surface workshop and offices.

## 18.8 Workforce Accommodation

### 18.8.1 Permanent Accommodation Camp

The accommodation capacity of the 650 man permanent camp has been determined as the estimated operations workforce plus a margin of 15%, rounded up to suit standard accommodation block sizes. The margin covers variability in estimates of workforce numbers, staff turnover, the requirements for visitors, and the need to accommodate extra personnel for maintenance shutdowns.

Accommodation will be provided for personnel as follows:

- Management personnel - executive units comprising a bedroom, living room and en suite bathroom. The units are located in duplex pairs. Nine duplex units will be provided to accommodate 18 people.
- Professional, sub-professional and mid management personnel - motel style rooms with en suite bathrooms provided in units of eight rooms each. A total of 29 units will be provided to accommodate 232 people.

- General staff – hostel style rooms will be provided in units of 16 rooms each with a communal ablution block at the end of each unit. A total of 25 units will be provided to accommodate 400 people.

The permanent camp will be located adjacent to the process plant and accessed from the main plant access road. The camp support facilities will include the following:

- Dry Mess: The kitchen and dining facility will include a dining area, kitchen, preparation areas, refrigeration and storage facilities.

- Wet Mess: The wet mess will be an open plan area housing a bar, TV area, games room, and a pool table / darts / cards area and ablutions.

- Laundry: The building will contain commercial washers and dryers. Laundering of bed linen, towels, etc. and the staff clothing will all be managed in this building by dedicated staff.

- Administration / Shop Building: The building will house a reception area, storage / supply room, shop and office space for the camp administration staff.

- Sporting Facilities: Gymnasium, swimming lap pool and multipurpose basket ball/tennis court.

### 18.8.2 Temporary Construction Accommodation

The existing exploration camp will be used for pioneer accommodation pending early completion of elements of the permanent camp.

Contractors will be required to provide their own accommodation for the construction work force and early mine development team. Accommodation will be arranged in temporary camps established by the contractors adjacent to the permanent camp location.

## 18.9 Other Facilities

Access to the plant, mine and accommodation village will be controlled by security check points. The plant and mine access will be via a common access road and security gate, with separate access and control for vehicles and pedestrians.

Facilities on the site will be fully fenced with an appropriate level of fencing to prevent access of wild animals.

A diesel storage facility and bowser will be provided in the main plant area for the plant area mobile vehicles.

A separate storage and refuelling facility will be provided at the mine services area for the refuelling of mine vehicles and equipment.

The communication system infrastructure will include the following:

- Mobile phone and internet linked to Zambia Telecom.
- Satellite television for the permanent camp.
- Local distribution via PABX and fibre link.
- Local radio network for operations and security communications.

# 19.0 MARKET STUDIES AND CONTRACTS

During the PFS, at a time when the project was forecast to produce a mix of flotation concentrates and cathode copper, Blackthorn commissioned Base Metals Marketing Services Ltd (BMMS) to produce a marketing report to examine the future market for Kitumba's production.

The report was provided in two parts, a Marketing Report for copper concentrates and an Addendum covering the copper cathode market.

With the flowsheet adopted for the OPFS Kitumba becomes a net purchaser of copper concentrates. As Kitumba will be competing with Zambian smelters for concentrates the BMMS report has been used as a basis for determining terms for the purchase of concentrates.

Based on current operations and those under construction in Zambia, the in-country concentrate balance suggests that output from mining operations will exceed domestic smelter capacity by the middle of the current decade.

Kitumba will be in a position to take advantage of the surplus of concentrates in the country to negotiate favourable terms for the purchase of concentrates to meet its operational requirements in the early years.

A preliminary estimate of concentrate requirements is shown in Table 19.1.

**Table 19.1 Concentrate Requirements by Year**

| | Year 1 | Year 2 | Year 3 | Year 4 | Year 5 | Year 6 | Year 11 | LOM Total |
|---|---|---|---|---|---|---|---|---|
| Concentrate, t | 32,057 | 39,417 | 39,391 | 34,677 | 16,507 | 9,820 | 23,974 | 195,844 |
| Contained Cu, t | 7,758 | 9,539 | 9,533 | 8,392 | 3,995 | 2,376 | 5,802 | 47,394 |

Concentrates purchased would ideally comprise predominantly chalcopyrite mineralisation with copper and sulphur contents of nominally 24 and 32 percent respectively. With concentrate freight costs based on the weight of concentrate, not the contained copper content, a domestic purchaser of this type of concentrate has a clear commercial advantage over the export market.

The BMMS report is a desktop study intended to provide a general overview and discussion of the likely future market for copper concentrates and cathode. As at April 2014 no direct contact has been made with concentrate producers or smelters or other potential buyers and therefore work to date does not constitute a detailed marketing study. This will be undertaken as part of a future definitive feasibility study.

The following provides an overview of the BMMS findings and how they have been used as inputs to the project Financial Assessment.

## 19.2 Copper Price Forecast

The key determinants for future copper prices are the forecast supply/demand balance for refined copper, developments in the industry cost structure as well as the activities of speculative investors.

Between 2016 and 2025 global Industrial Production is forecast to grow by around 4.3% pa In line with the past positive link between global economic growth and copper consumption growth, BMMS expects this positive correlation to be maintained going forward. Copper demand is therefore forecast to rise by around 3.3% pa for the same period, while refined copper production is forecast to rise by a slightly lower 3.1% pa.

Although on paper there are sufficient projects to satisfy demand, it remains to be seen whether they are able to be brought on stream within their planned timescales given all the above challenges. As a result, the balances in any one year can still change materially. If significant delays in bringing on new near-term capacity occur, then the forecast surplus to 2016, could be short lived.

Copper ore grades are forecast to fall further, energy costs are forecast to rise and project CAPEX has also been on the increase. The industry cost curve is expected to continue to rise, which will result in the requirement for a higher copper floor price.

For the period 1989 – 2012 the compounded annual growth rate (CAGR) of mine operating costs amounted to around 3.0% pa. Assuming a continuation of this trend, by 2016 full cash costs would rise to $3.15/lb and by 2020 to $3.55/lb.

As advised by Blackthorn, a long term copper price of US$3.50/lb has been used for the project Economic Analysis based on information sourced from Wood Mackenzie, Sydney.

## 19.3 Concentrate Purchases and Cathode Sales

The flowsheet adopted for OPFS converts all copper to cathode with no sale of concentrate. However, in the early years the project will have to purchase suitable sulphide concentrate and process it through the POX autoclave to generate acid and heat for the process. Terms for the purchase of concentrate are based on the BMMS report with some updates and adjustments as laid out below.

The January 2014 spot TCRC for concentrates delivered to China were quoted as $100 - 105 per tonne and 10 - 10.5 cents per pound for standard copper concentrates. This is significantly in excess of the $77.50 per dry tonne and 7.75 cents per pound quoted in the BMMS report. The current quoted TCRC terms were used as the basis for pricing purchased concentrates.

Although the global copper concentrate market has an influence on concentrate sale / purchase terms local sales terms are influenced by the high cost of exporting concentrates and particularly by the imposition of an export tax which commenced in 2008 (currently 10% of the concentrates value).

Zambian smelters charge a so-called 'Freight Realisation' or 'Captive Freight' charge. This is a participation in the freight advantage of being able to deliver to domestic smelters versus having to send the concentrates to overseas destinations. The smelters' logic is that they have the burden of higher freight cost for the resulting refined metal, as the ultimate consumption of copper will not be in Zambia.

For the purposes of the Economic Analysis it has been assumed that the cost of concentrate export freight, which would be to the account of the seller, is deducted from the price paid for domestic concentrates purchased by Kitumba. In addition it has been assumed that the equivalent cost of the export tax is deducted from the price paid for domestic concentrates purchased by Kitumba. OPFS copper concentrate purchase terms are summarised in Table 19.2.

**Table 19.2 OPFS Copper Concentrate Purchase Terms**

| | |
|---|---|
| Copper Price | $3.50/lb |
| Delivery Terms | FIS Kitumba |
| Payable Copper | 96.5% of contained copper with a minimum deduction of 1% |
| Treatment Charge | $100/dry tonne concentrate |
| Refining Charge | $0.10/lb payable copper |
| Freight Realisation Charge | $250/dry tonne concentrate |
| Export Tax Charge | $187/dry tonne concentrate |
| Net Price Paid (for 24.2% Cu conc.) | $1,200/t concentrate<br>$2.25/lb contained copper |
| Payment | 30 days after receipt |

Taking into account the flow of current exports it is likely that Kitumba cathodes will follow the same trend with their cathodes being sold to consumers in China and other Asian countries.

For the purposes of the Economic Analysis it has been assumed that Kitumba will export to overseas customers with payment received upon arrival in the customer's warehouse 30 days following dispatch.

Cathode sales terms are summarised in Table 19.3.

**Table 19.3 OPFS Copper Cathode Sales Terms**

| | |
|---|---|
| Delivery: | CIF liner terms Chinese port |
| Copper payment: | LME settlement price (assumed to be $3.50/lb) |
| Premium: | Nil |
| Payment: | 100% paid 30 days after dispatch from site |
| Freight Rate: | $214/t (normalised costs to China) |
| Transport Insurance: | Included in above |

No allowance has been made for premium payments commonly paid for cathode over the LME price on the basis that premiums received will cover marketing and / or trader profit margins.

# 20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

## 20.1 Preparation and Submission of EIS

Africa Geo-Environmental Services Gauteng (Pty) Ltd (AGES) has investigated the feasibility of the proposed the project from an environmental and social impact management perspective and facilitated the environmental impact assessment processes required under Zambian law and in accordance with the principles of sustainable development.

An Environmental Project Brief (EPB) was submitted to the Environmental Council of Zambia (ECZ) which is now known as the Zambia Environmental Management Agency (ZEMA) on 22 April 2010. The EPB was approved in a letter dated 28 May 2010. Prospecting activities are under way on the project site in accordance with the conditions of the approval and all other relevant environmental legislation.

Following the EPB a draft Terms of Reference (ToR) for the Environmental Impact Assessment (EIA) Study was prepared after a consultation / scoping meeting with ZEMA held on 28 November 2012 and submitted to ZEMA on 13 December 2012. On 15 January 2013, ZEMA requested amendments to the ToR. The updated ToR was re-submitted to ZEMA on 22 January 2013. On 23 January 2013 the ToR was approved by ZEMA, who instructed that the project may continue to the EIA phase.

AGES prepared a draft Environmental Impact Study (EIS) in accordance with the approved ToR. The Draft EIS was submitted to ZEMA on 12 September 2013. Comments on the EIS were received from ZEMA on 7 October 2013. The EIS was subsequently updated with the received comments and re-submitted to ZEMA on 12 December 2013. On 31 January 2014, ZEMA confirmed that they had reviewed the second draft of the EIS and observed that it addressed the majority of the environmental concerns raised from the first draft review. ZEMA subsequently stated that they had no objection to the submission of the final reports and on 24 March 2014, the required 12 copies of the final EIS were submitted to ZEMA.

ZEMA authorisation for the proposed mining activities in terms of the relevant environmental legislation is currently pending.

AGES completed an environmental baseline investigation of the project site and its surroundings during 2012. The purpose of the baseline assessment was to obtain a better understanding of the broader project context in order to facilitate and expedite subsequent processes associated with the project, along with the identification of any potential adverse project parameters so that these could be addressed early in the project planning.

The AGES work programmes evaluated the existing socio-economic and biophysical environments of and around the proposed project site in order to:

- Determine whether ecological and/or socio-economic conditions exist on or around the proposed development site which may impact (positively or negatively) on the proposed project
- Determine if and how the proposed project will impact on the existing environmental (ecological and/or socio-economic) conditions on the site and surrounds
- Guide the project team on environmental legal requirements and authorisations that must be addressed in the development of the project
- Identify opportunities associated with the project for environmental (socio-economic and ecological) improvement, and provide benchmarking and environmental best practice standards to be incorporated into the project design
- Provide guidance on the proposed way forward for the project in terms of environmental management considerations and applicable environmental legislation, as well as to suggest measures to optimise the project from an environmental management perspective.

## 20.2 Ongoing Activities

AGES compiled an environmental report (refer Section 3.0) to specifically address environmental-legal requirements applicable to the project, environmental risks presented to the project, and the relevant way forward from an environmental impact management perspective.

AGES is assisting Blackthorn Resources in the process of applying for environmental authorisation, water use permissions and a mining right for the project. Consultation with ZEMA will be instrumental in streamlining the applications for water use permissions and a mining right. Further requirements for stakeholder engagement will also be determined by ZEMA.

No fatal flaws have been identified in terms of environmental management considerations, although a number of follow-up and additional studies to be conducted as part of the DFS phase of the project have been recommended in the AGES report. These include the following:

- Additional hydrogeology testwork and investigation.
- Geochemistry and water quality.
- Surface water and stormwater management.
- Ecology, soils and land use potential and wetlands.
- Socio-economic, historical and cultural aspects.

- Traffic.
- Air quality.
- Visual impact.
- Noise.
- Rehabilitation and financial provision.
- Additional specialist studies including quantification and mitigation of risks associated the uranium present.

Environmental authorisation for the project is pending, along with water use licensing and the required mining right. The project team is in on-going consultation with the respective authority representatives in order to facilitate applications under the abovementioned regulatory requirements.

Blackthorn submitted an application for a large scale mining license in July 2014.

# 21.0 CAPITAL AND OPERATING COSTS

## 21.1 Capital Cost

The overall OPFS capital cost estimate was compiled by Lycopodium. The capital cost estimate reflects the project scope as described in the OPFS report. Mine capital costs were developed by AMC and are included in the estimate table below.

KP provided quantities for a number of items including the tailings storage facility and water storage dam with rates provided by Lycopodium to derive the capital estimate.

All costs are expressed in US dollars ($) unless otherwise stated and based on 1Q2014 pricing. The estimate is deemed to have an accuracy of ±25%.

The various elements of the project estimate have been subject to internal peer review by AMC and Lycopodium and have been reviewed with Blackthorn for scope and accuracy.

The capital estimate is summarised in Table 21.1. The initial project capital cost was estimated at $680.3 million and the maximum cash drawdown to a position where cash flow is positive was estimated to be $696.7 million.

**Table 21.1 Initial Capital Cost Estimate Summary (US$, 1Q2014, ±25%)**

| Main Area | Initial Capital ($M) |
|---|---|
| Mining[1] | 107.8 |
| Construction Indirects | 26.7 |
| Treatment Plant | 266.1 |
| Reagents and Services | 27.3 |
| Infrastructure & Tailings | 67.3 |
| Owners Costs | 42.8 |
| EPCM Costs | 58.8 |
| Contingency | 83.5 |
| **Project Total** | **680.3** |

**[1]Includes AMC capital estimate and accommodation and meals during construction**

The estimate is subject to the following qualifications:

- All labour rates, materials and equipment supply costs are current at 1Q14. Contingency has been allowed based on the quality of the information, but there is no allowance for escalation in the estimate.
- Construction contractor rates include mobile equipment, vehicles, fuel, construction power and consumables for the duration of construction. Potable water and raw water supply will be provided by the client and available at site for the use by contractors.

- Accommodation, meals and mobilisation / demobilisation / R&R flights of construction contractor personnel are incorporated in the contractor indirect labour rates on the basis of individual contractors making their own accommodation arrangements.

- Meals and accommodation for the owner and EPCM contractor teams have been allowed in the estimate.

- Project spares are a percentage (6%) allowance of the mechanical supply cost based on similar size projects.

- A commissioning assistance crew is allowed for in the EPCM allowance.

- PLC programming for the process plant has been allowed for in the EPCM allowance.

- Site supply of power, supply of raw water (for operations and construction), sewage removal and treatment, communications network for construction facilities are included in the infrastructure costs.

The following is excluded from the overall project capital costs:

- Working capital (this is however addressed in the Cash Flow Model).

- Duties / taxes / fees.

- Project sunk costs (allowance made in the Cash Flow Model).

- Project escalation.

### 21.1.2 Deferred / Sustaining Capital

Deferred / sustaining capital for ongoing mine development was estimated by AMC and has been included in the Cash Flow Model used for the Economic Analysis.

Similarly an allowance of $3 million per year ($1/t) has been made for progressively increasing the capacity of the tailings storage facility.

Sustaining capital for the process plant and surface infrastructure is included in the factored maintenance costs used in the operating costs estimate (some $10.25 million per year excluding labour and crusher and mill liners).

## 21.2 Operating Costs

The overall OPFS operating cost estimate was compiled by Lycopodium. Mine operating costs were developed by AMC and are included in the estimate tables below.

Operating cost estimates are considered to have an accuracy of ±25% and are presented in US dollars ($) based on prices obtained during the first quarter of 2014 (1Q14).

The total operating costs for the Kitumba Copper Project are summarised in Table 21.2.

**Table 21.2 Operating Cost Summary (1Q14, ±25%)**

| Cost Centre | LOM Cost, $M | $/t Ore | $/t Cu | $/lb Cu |
|---|---|---|---|---|
| Mining | 806.7 | 25.53 | 1,266 | 0.57 |
| Surface & Infrastructure | | | | |
| Operating Consumables | 764.6 | 24.20 | 1,200 | 0.54 |
| Maintenance Materials | 109.7 | 3.47 | 172 | 0.08 |
| Labour | 30.2 | 0.96 | 47 | 0.02 |
| Power | 283.0 | 8.96 | 444 | 0.20 |
| Laboratory | 19.4 | 0.62 | 31 | 0.01 |
| General and Administration | 52.9 | 1.67 | 83 | 0.04 |
| Product Transport | 136.1 | 4.31 | 214 | 0.10 |
| **Total** | **2,202.7** | **69.71** | **3,457** | **1.57** |

A summary of the fixed and variable components of the life of mine operating costs for each cost centre is presented in Table 21.3 below. Variable processing costs are those costs which are directly proportional to the throughput rate, ore composition, and metal production, while fixed costs are considered to be those which will not change for up to 15% variation around the base ore throughput rate (3 Mtpa). If the throughput varies by more than ±15%, then changes to the fixed costs may be brought about by:

- Variation in the number of operating and maintenance personnel.
- Variation in the plant utilisation (operating time).
- Changes to equipment size (power draught).

**Table 21.3 Summary of Fixed and Variable Operating Costs (1Q14, ±25%)**

| Cost Centre | Fixed Costs (LOM) | | Variable Costs (LOM) | | Total Processing Cost (LOM) | |
|---|---|---|---|---|---|---|
| | **$M** | **$/t Ore** | **$M** | **$/t Ore** | **$M** | **$/t Ore** |
| Mining | 564.7 | 17.87 | 242.0 | 7.66 | 806.7 | 25.53 |
| **Surface & Infrastructure** | | | | | | |
| Operating Consumables | 32.8 | 1.04 | 731.8 | 23.16 | 764.6 | 24.20 |
| Maintenance Materials | 34.5 | 1.09 | 75.2 | 2.38 | 109.7 | 3.47 |
| Labour | 30.2 | 0.96 | - | - | 30.2 | 0.96 |
| Power | 34.6 | 1.10 | 248.4 | 7.86 | 283.0 | 8.96 |
| Laboratory | 3.3 | 0.11 | 16.1 | 0.51 | 19.4 | 0.62 |
| General and Administration | 52.9 | 1.67 | - | - | 52.9 | 1.67 |
| Product Transport | - | - | 136.1 | 4.31 | 136.1 | 4.31 |
| **Total** | **753.2** | **23.83** | **1,449.6** | **45.88** | **2,202.7** | **69.71** |

The processing and administration cost estimates presented in the tables are exclusive of the following:

- Project financing or interest charges.
- Escalation and impact of exchange rate fluctuations.
- Customs and import duties and VAT.
- Exploration costs.
- Rehabilitation costs.
- Land tenure and tenement fees.
- Local council rates, subsidies, fees and environmental fees.
- Royalty payments.
- All costs associated with areas beyond the battery limits of the Study.

### 21.2.2 Power

Electric power consumption for the process plant and surface infrastructure has been calculated based on the installed power from the mechanical equipment list, with drive efficiency factors and equipment utilisation applied as appropriate. A unit cost of power of $0.095/kWh, supplied from the ZESCO grid, was used to calculate process plant and infrastructure power costs.

Mine power costs are included in the overall mine operating costs. Mine power demand is approximately 2.5 MVA for most of the mine life but peaks at around 5.4 MVA during the early stages of mine development prior to commissioning the process plant.

The average continuous power draw of 38.5 MW for the process plant and surface infrastructure includes the 'free issue' of power for the oxygen plant. A peak mine power draw of approximately 5 MW occurs post-commissioning resulting in a peak average continuous site power draw of approximately 45 MW and a more usual draw below 42 MW.

### 21.2.3 Concentrate Purchases

Although not strictly classified as a reagent, imported flotation concentrate will play a major role in addressing the chemical requirements within the process plant and will be a major contributor to the overall operating cost. The cost to the project of importing flotation concentrate from a source in Zambia has been based on the criteria shown in Table 21.4. This assumes that the concentrate will be predominantly a chalcopyrite flotation concentrate with a 24.2%$^w/_w$ contained copper content.

**Table 21.4 Concentrate Pricing Criteria**

| Description | Value |
|---|---|
| Concentrate Cu Content | 24.2%$^w/_w$ |
| Copper Metal Price | $3.50/lb |
| Metal Payment | 23.2% (24.2% plus 1% deduction) |
| Treatment Charge | $100/t of dry concentrate |
| Refining Charge | $0.10/lb of payable copper |
| Freight Realisation Charge | $250/t of dry concentrate |
| Export Tax Benefit | $187/t of dry concentrate (10% of Cu value) |
| **Total Cost of Concentrate** | **$1,200/t of dry concentrate** |

It will be required to import flotation concentrate for seven of the eleven years of the mine life. A summary of concentrate import requirements for each of these years is presented in Table 21.5.

**Table 21.5 Concentrate Import Summary**

| | Year 1 | Year 2 | Year 3 | Year 4 | Year 5 | Year 6 | Year 11 | LOM |
|---|---|---|---|---|---|---|---|---|
| Concentrate, t | 32,057 | 39,417 | 39,391 | 34,677 | 16,507 | 9,820 | 23,974 | 195,844 |
| Concentrate, $M | 38.5 | 47.3 | 47.3 | 41.6 | 19.8 | 11.8 | 28.8 | 235.0 |

### 21.2.4 Product Transport

An indicative cost of transport of cathode to market has been advised by Antrak Logistics based on flat deck trailer road haulage from site to the Tanzanian port of Dar es Salaam, loading into shipping containers, and shipment to a port in China (nominally Shanghai). A cathode copper unit transport cost of $213.64/t has been used (based on a 22 tonne payload per shipping container), a breakdown of which is summarised in Table 21.6.

**Table 21.6 Summary of Product Transport Cost**

| Cost Centre | Unit | $ | $/t |
|---|---|---|---|
| Zambia to Dar es Salaam, Tanzania – road | 1 t | 125 | 125.00 |
| Zambia to Dar es Salaam, Tanzania – FOB | 1 t | 45 | 45.00 |
| Shipping Container – Dar es Salaam to Shanghai | 22 t | 400 | 18.18 |
| Security Surcharge | 22 t | 10 | 0.45 |
| Bill of Lading | 22 t | 75 | 3.41 |
| Destination Terminal Handling Charge | 22 t | 250 | 11.36 |
| Container Loading and Re-delivery | 22 t | 150 | 6.82 |
| Agency / Documentation / Handling Fees | 22 t | 75 | 3.41 |
| **Total** | | | **213.64** |

# 22.0 ECONOMIC ANALYSIS

## 22.1 Variation from OPFS Evaluation

The mine schedule developed for the OPFS included a small quantity of material (139 kt with 918 t contained copper) which is classified as Inferred Resources. NI 43-101 Rules and Policies (Section 2.3(1)(b)) require that 'An issuer must not disclose the results of an economic analysis that includes or is based on inferred mineral resources…' therefore the material falling into the inferred category has been removed from the processing schedule and the Cash Flow Modelling undertaken for the OPFS repeated for the purposes of this Technical Report. The re-classification of this material as waste rather than mill feed does not materially affect the proposed mine plan or mining costs.

## 22.2 Basis of Economic Analysis

The economic analysis of the project was conducted using a simple cash flow model prepared by Lycopodium on behalf of Blackthorn. The model was structured using an Excel workbook.

Input data came from a variety of sources, including the original PFS study, Blackthorn and other members of the OPFS team. The economic analysis was based upon:

- Capital cost estimates and expenditure schedules prepared by Lycopodium and AMC.
- Mine schedule and mining operation cost estimates based on the OPFS strategy of the underground mining operations being contractor-operated, as developed by AMC.
- Process operating and general and administration costs estimates prepared by Lycopodium, with contributions from Blackthorn and other members of the OPFS team.
- Metallurgical performance characterised by testwork conducted on composite samples from the Kitumba deposit.
- A sustaining capital cost estimate for the mining operation prepared by AMC.
- A sustaining capital cost allowance for the tailings dam.
- Owners capital cost estimates prepared by Lycopodium and Blackthorn.
- Royalty, tax, discount rates and other model inputs provided by Blackthorn.
- The cash flow analysis excludes any effects due to inflation and all dollars are expressed as real United States dollars as at 1Q2014.
- The forecast copper price of $3.50/lb for the life of mine is based on the long term copper price, provided by Blackthorn and sourced from Wood Mackenzie, Sydney.

- The cash flow analysis is based on full equity funding.

## 22.3 Outcomes

The cash flow model reports:

- All costs in real US Dollars ($) exclusive of escalation or inflation.
- A net present value (NPV) at an 8% discount rate; and
- An internal rate-of-return (IRR) based on after-tax net cash flows.

The outcomes are summarised in Table 22.1.

**Table 22.1 Project Economic Analysis Summary**

| | | |
|---|---|---|
| Revenue from copper (based on $3.50/lb) | 4,909.4 | $M |
| Total cash cost excluding royalties (C1) | 1.57 | $/lb Cu |
| Total cash cost (including royalties) | 1.81 | $/lb Cu |
| All-in cost * | 1.89 | $/lb Cu |
| Capital expenditure (Life-of-Mine) | 796.2 | $M |
| Initial capital investment (excl working capital) | 680.3 | $M |
| Peak funding | 696.8 | $M |
| Deferred and sustaining capital | 115.9 | $M |
| Plant and equipment salvage | 50.0 | $M |
| Closure cost | 33.5 | $M |
| **Pre-Tax Economics** | | |
| Free cash flow after cost allocation (undiscounted) | 1,628.6 | $M |
| Internal rate of return (IRR) | 25.3 | % |
| Project NPV (discounted at 8.0%) | 694.7 | $M |
| Payback period | 3.4 | Years |
| **After-Tax Economics** | | |
| Free cash flow after cost allocation (undiscounted) | 1,202.0 | $M |
| Internal rate of return (IRR) | 21.2 | % |
| Project NPV (discounted at 5.0%) | 458.7 | $M |
| Payback period | 3.5 | Years |

*** Total cash cost, including sustaining and deferred capital.**

## 22.4 Assumptions and Qualifications

The cash flow analysis is based on the following:

Basis of Estimate

- The cash flow model has been based on a 27 month project development period, assuming the cash-outflows commence in Year -3 and that copper production commences in Month 1 of Year 1. The model has considered only cash flows from project 'go-ahead', with all previous expenditure considered tax losses carried forward to the start of production.

- Monthly mined tonnage and head grade have been based on the mining schedule and process plant throughput and copper production rates as presented in the OPFS with all Inferred material removed from the schedules.

- The mining, processing and administration costs are based on the operating cost estimates presented in the OPFS.

- The overall copper recovery figure of 92.3% is based on testwork and interpretation.

- The capital costs are based on the estimates presented in the OPFS. Scheduling of capital expenditure was based on typical expenditure s-curves for a project of this size and type.

- Provision was made for closure and rehabilitation costs of $33.5 million payable in the last month of production and a plant and equipment salvage value of $50 million.

- The treatment of depreciation and company taxes are based on Blackthorn's understanding of current Zambian tax law.

Depreciation

- It has been assumed all capital expenditure is depreciated and amortised over four years on a straight line basis. All tax losses relating to the operation during ramp-up and depreciation and amortisation charges are carried forward and utilised when the project has sufficient profits. This is consistent with guidance provided to Blackthorn by KPMG.

- In the final year of operation, the residual capital allowance is depreciated against the revenue (including the salvage value of the plant).

Company Tax

- Provision has been made for company tax at 30% of gross profit. This is the rate of corporate tax in Zambia.

- Tax losses are carried forward over a period of up to 10 years from the year in which the loss was incurred.

- It has been assumed that $40 million from costs sunk to date is available to be carried forward at the start of production have been accrued by the project in Zambia, as advised by Blackthorn. This amount is not included in the capital cost estimate.

Copper Price

- A copper price of $3.50 per pound has been applied in the cash flow model. This forecast copper price is based on the long term copper price sourced from Wood Mackenzie, Sydney

- Transport costs from site to China via Dar-es-Salaam have been included in the operating cost estimate.

- No further marketing or refining and treatment costs have been applied to the copper cathode price.

Concentrate Payment Terms

- During the first six years and the last year of production, the project will be processing imported copper concentrate in addition to the concentrate derived from Kitumba ore. The payment terms adopted for the purchase of concentrates are provided in the OPFS.

Royalties

- Government royalties are allowed at 6.0% of gross revenue from copper mined at Kitumba.

- BHPB NSR royalties are allowed at 2.0% of revenue from copper mined at Kitumba after realisation cost. The realisation cost includes the processing cost, transport cost and government royalty, but excludes mining and general and administration cost.

General

- The cash flow model assumes full equity funding.

- No provision has been made for interest on cost of capital.

- No provision has been made for corporate head office costs during operations.

- No provision has been made for escalation or inflation.

- No Value Added Tax (VAT) payable.

- No provision has been made for additional taxation or costs related to the repatriation of funds from Zambia.
- The NPV calculation is based on payments occurring at the end of each month.
- Working capital has been excluded from the capital cost estimate, but implicitly included in the cash flow model as the difference between the peak funding requirement and the initial capital expenditure.

## 22.5 Sensitivities

The sensitivity of key project financial parameters to changes in copper price, capital and operating costs and copper recovery are shown in Figures 22.1 to 22.2.

**Figure 22.1 Sensitivity of IRR**

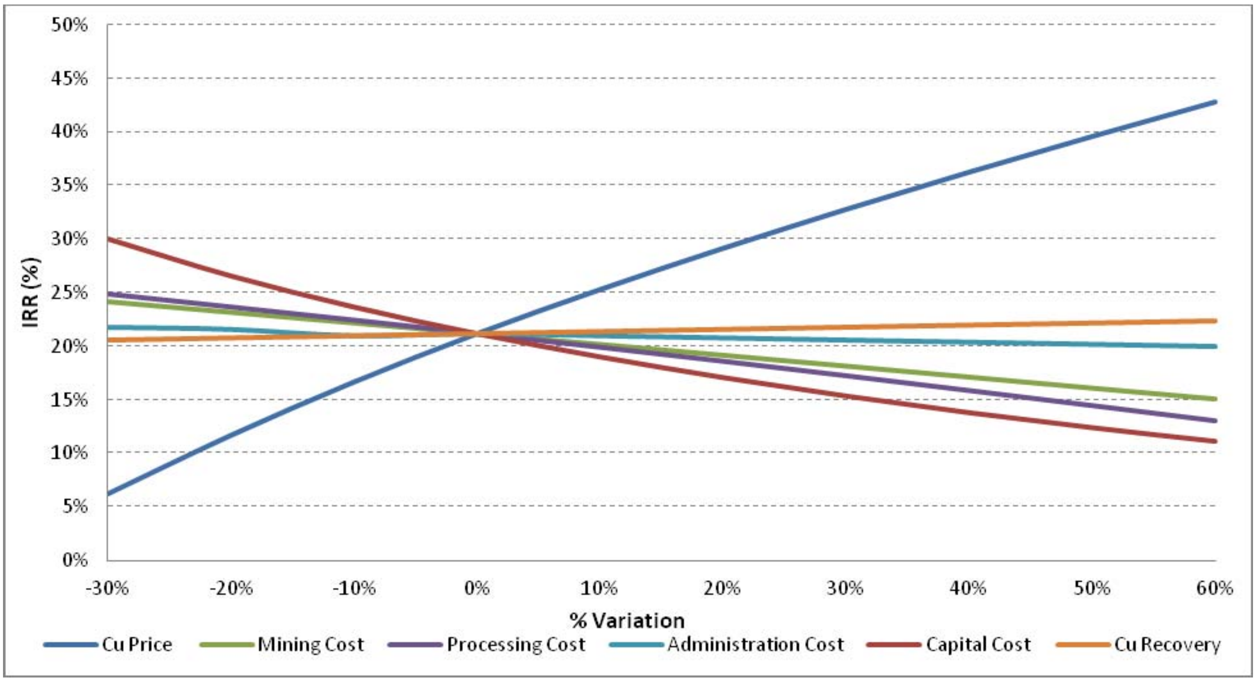


**Figure 22.2 Sensitivity of NPV**

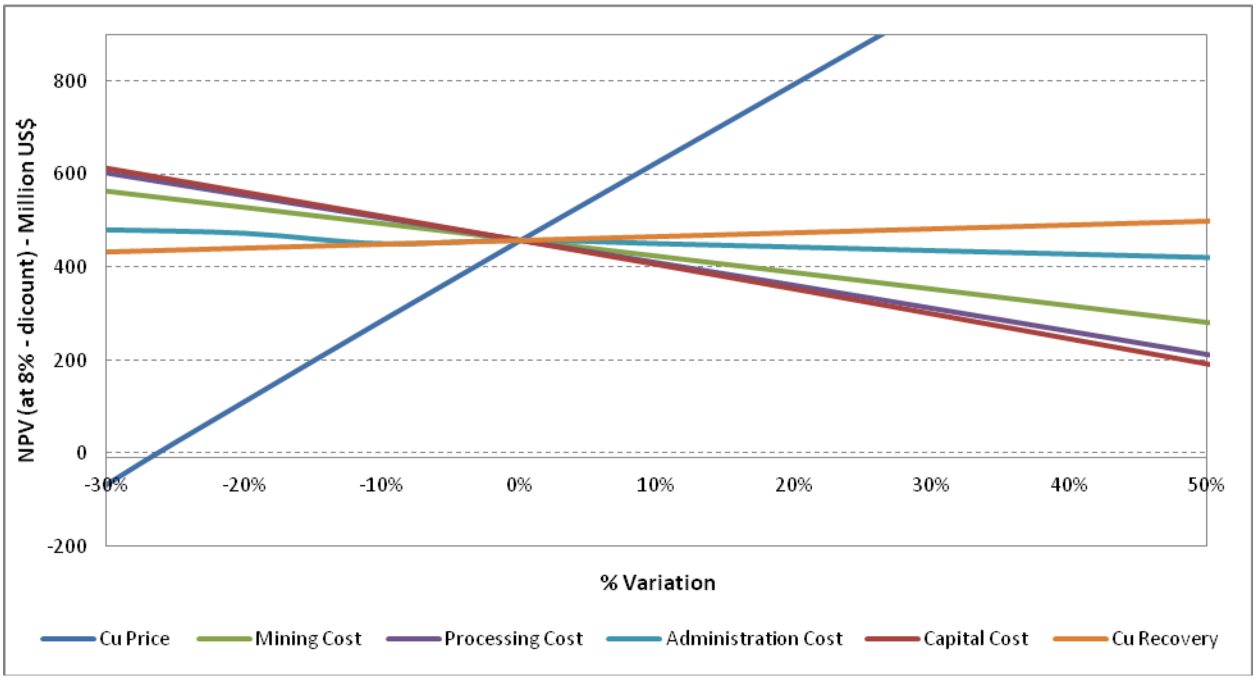
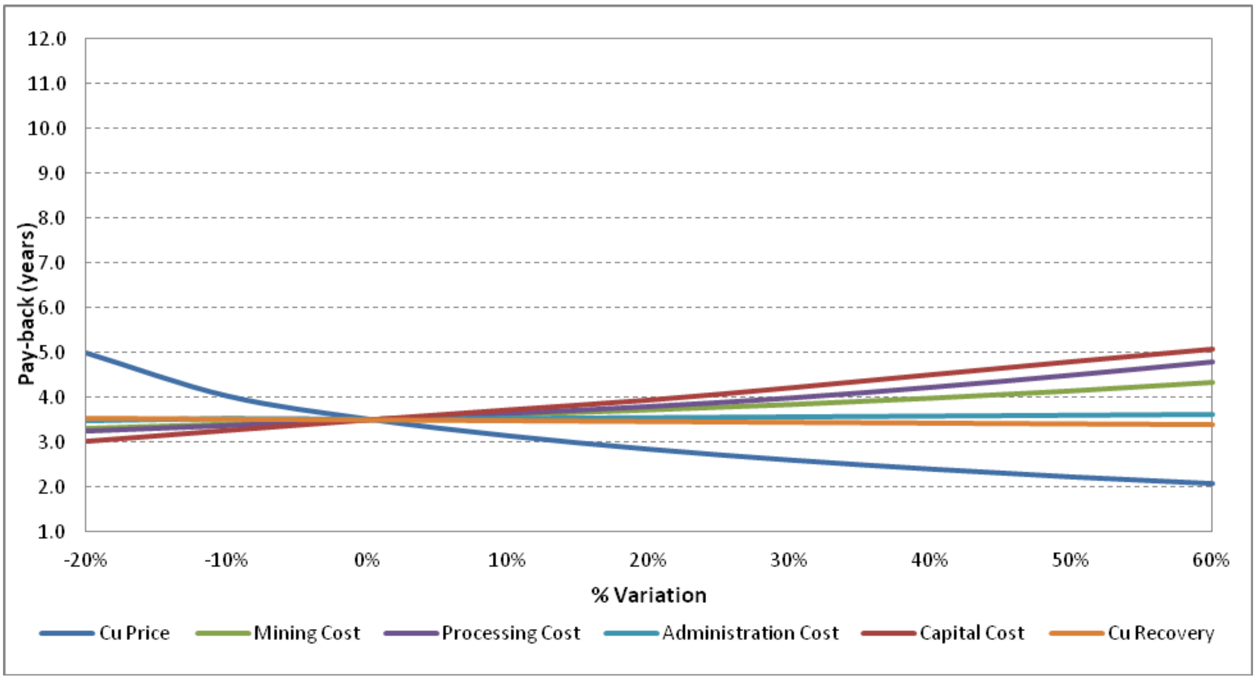



**Figure 22.3 Sensitivity of Payback Period**



# 23.0 ADJACENT PROPERTIES

Not relevant.

# 24.0 OTHER RELEVANT DATA AND INFORMATION

## 24.1 Project Implementation

The approach to project implementation outlined below was used as the basis for the preliminary implementation schedule and the build up of the capital cost estimates. The approach to be adopted for project implementation will be developed further during the definitive feasibility study with the preparation of a preliminary Project Execution Plan (PEP).

The Owner's team will be progressively expanded to widen its skills and knowledge base to meet the needs of the Project. The Owner's team will manage both the onshore and offshore activities of the principal EPCM contractor and specialist subcontractors as well as providing specialist technical input into the project design.

Key onshore mine operations roles will be filled early to contribute to mine design and manage the mine contractor undertaking development of the boxcut, decline, mine production levels and underground infrastructure.

An expanded Blackthorn Zambia management, administration and services department will manage environment and community issues and prepare the site for the coming influx of operating personnel.

Prior to embarking on project implementation, additional site works, drilling and sampling, laboratory testwork and investigations and collection of data will have to be undertaken to fill gaps in the existing knowledge base. The information generated from these activities will be combined with the output of existing and additional engineering and estimating work to complete a DFS.

As the development of the boxcut, portal, decline, production levels and underground infrastructure is on the critical path for achieving earliest possible copper production, the early works will be undertaken by an experienced underground mining contractor under the direction of senior Kitumba mine operations personnel.

Particularly during the early years the site mining team will be supported by consultants providing geotechnical, mine design, hydrogeology and other specialist knowledge and support.

The decision to engage a mining contractor has been made as a contractor will have immediate access to skilled and experienced personnel as well as access to the equipment required to undertake and support the development works. They will also be in a position to mobilise, at short notice, additional personnel and materials to meet unexpected demands that arise, and to keep the development works on schedule. The long term mine operating costs developed for the study have assumed contract mining will be retained.

For the process plant and surface infrastructure, the implementation strategy on which the OPFS is based is an EPCM approach where an Engineer (an internationally accredited engineering company) will be responsible for managing all aspects of the construction activities under the broad direction of an owner's team.

A preliminary Project Execution Plan (PEP) incorporating a schedule and control budget will be developed for the definitive feasibility study outlining the overall management methodology for the delivery of the Project, including engineering, procurement, construction, commissioning and handover.

The PEP will include strategies for all aspects of project management and control across all the project functions and phases.

The preliminary schedule developed for the OPFS allowed 12 months to complete activities related to the preparation and issue of a definitive feasibility study. This was to be followed by six months of front end engineering and design focused on long lead procurement items, tender and negotiation of early contracts and a Heads of Agreement with ZESCO for power supply.

Once full project 'go-ahead' was approved, a duration of 25 months was estimated to first ore to mill. Ramp up to 75% production was estimated to take three months with a further six months to reach nameplate capacity (refer Section 24.2).

Several items and activities are potentially on or close to the schedule critical path for design and construction including mine development and supply and installation of long lead mechanical and electrical equipment items for the process plant. Their criticality should be investigated further during the DFS.

The establishment of the power supply from the Zambian national grid while not currently identified as a critical path item has the potential to influence the schedule as it is dependent on activities not completely within the projects control. This has been identified as a potential risk to the schedule. Potential sources of delay include establishment of a suitable easement for the power line, completion of commercial negotiations with ZESCO as well as subsequent technical approvals and agreements regarding responsibility for tendering and managing the supply and installation of the HV power line and switchyards.

## 24.2 Operations

The current operating cost estimate is based on a preliminary organisational structure that is considered to be appropriate for an operation of this scale and type in the region.

The preliminary organisational structure will be reviewed during the DFS and finalised during the initial stages of project implementation when senior operating personnel will take ownership of creation of the operating teams.

The project is committed to providing employment opportunities to Zambian nationals, based on identifying individuals with the necessary qualifications, experience and skills. It is anticipated, however, that expatriate staff will fill a significant number of senior management, key supervisory and training roles. A succession plan will be developed to transition Zambian nationals into roles initially filled by expatriates.

Zambia has a long history of commercial copper mining and it is expected that many key management and supervisory roles can be filled from the national pool of experienced personnel.

It is recognised that the site is remote from the main mining areas and even the nearest local town of any appreciable size is a 100 km round trip from the site. Allowance has, therefore, been made to employ most Zambian nationals on a single status residential 'bus in / bus out' basis. Employment opportunities will be offered to local residents where feasible and appropriate.

Mine development and operations will be undertaken by an experienced mine contracting company under the direction of an owners technical and operations management team assisted, as required, by external consultants and service providers.

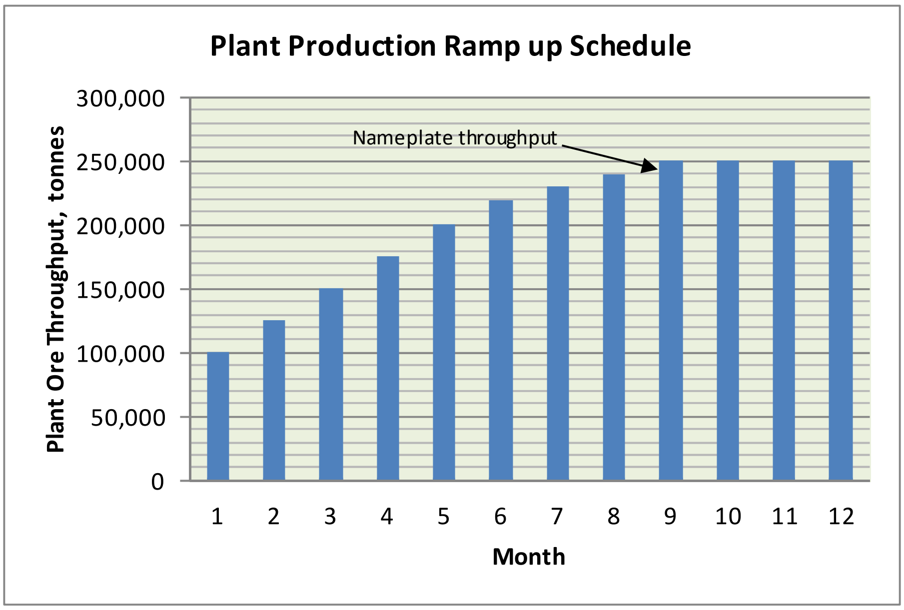
Several contract mining companies are already established in Zambia and/or elsewhere in the region and it is expected that they will tender for the work and be in a position to mobilise the initial team and supplementary skilled personnel and equipment as required to ensure that productivity and progress meet the project requirements.

Surface operations and administration roles will predominantly be filled by Kitumba staff. Exceptions to this include the operation and maintenance of the cryogenic oxygen plant, which will be supplied and operated under a Build Own Operate contract, and the maintenance and operation of the accommodation camp which will be contracted to a specialist service provider. Consideration will be given to contacting other non-core functions such as site security at an appropriate time.

Mining and processing plant ramp up and production estimates have been prepared to facilitate the development of a financial cash flow model and thus more accurately determine the timing of likely expenditure and revenue streams. The generation of limited quantities of ore from mining development activities will commence 12 months prior to ore being fed to the processing plant. During this pre-production period, the development ore will be stockpiled for later treatment and waste will be placed on a prepared waste rock dump or used for construction purposes.

The plant ramp up schedule has been developed to reflect the complexity of the processing facility, using ramp up data from similar commercial operations as a basis. It is estimated that nameplate plant ore throughput (250,000 t/month) will be achieved nine months after ore is first introduced. The first month will process an estimated 100,000 tonnes of ore, increasing at the rate of 25,000 tonnes per month over the next five months. Month 6 will process 220,000 tonnes, increasing by 10,000 tonnes per month for the next three months, until nameplate ore throughput is achieved at the end of Month 9. The process plant ramp up schedule is illustrated in Figure 24.1.

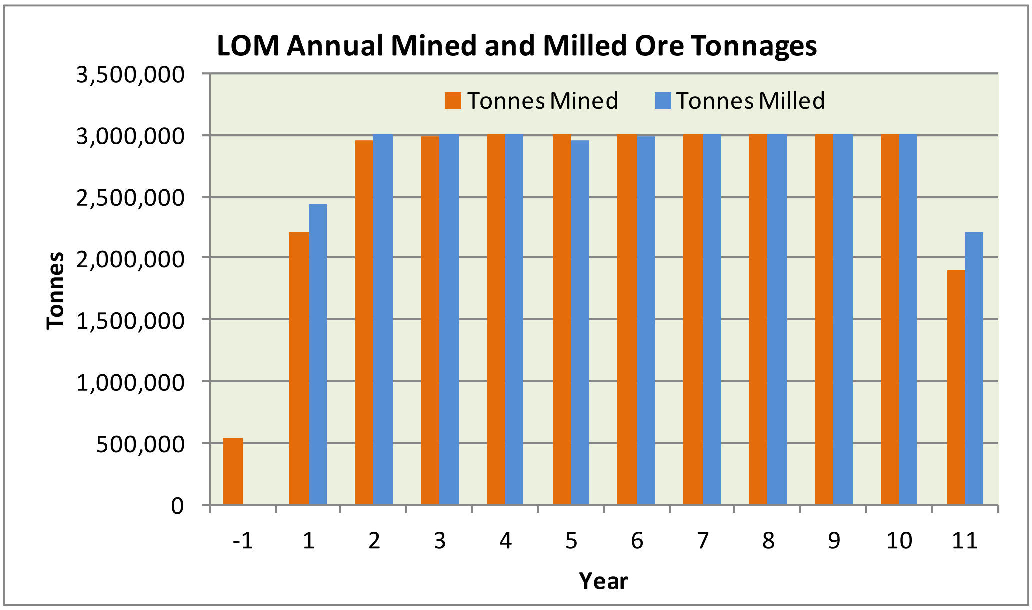
**Figure 24.1 Plant Production Ramp up Schedule**




The LOM mine production will be 31.6 Mt. Ore mining will commence one year prior to ore being fed to the processing plant, i.e. mining will start in Month -11 and plant feed will commence in Month 1.

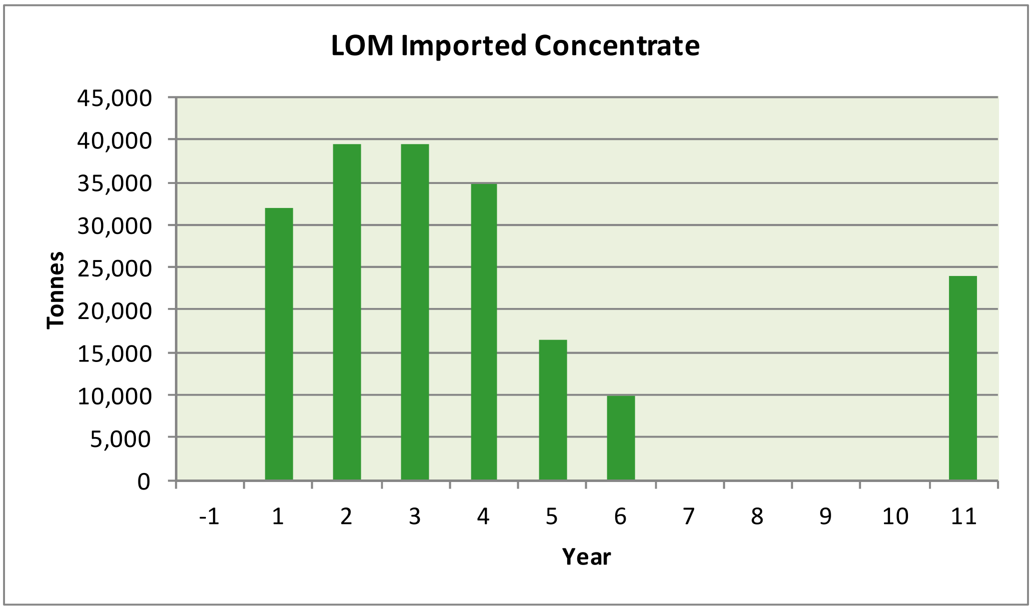
An annual comparison of the LOM tonnage mined and tonnage milled is shown in Figure 24.2.

**Figure 24.2 LOM Annual Ore Production Schedule**



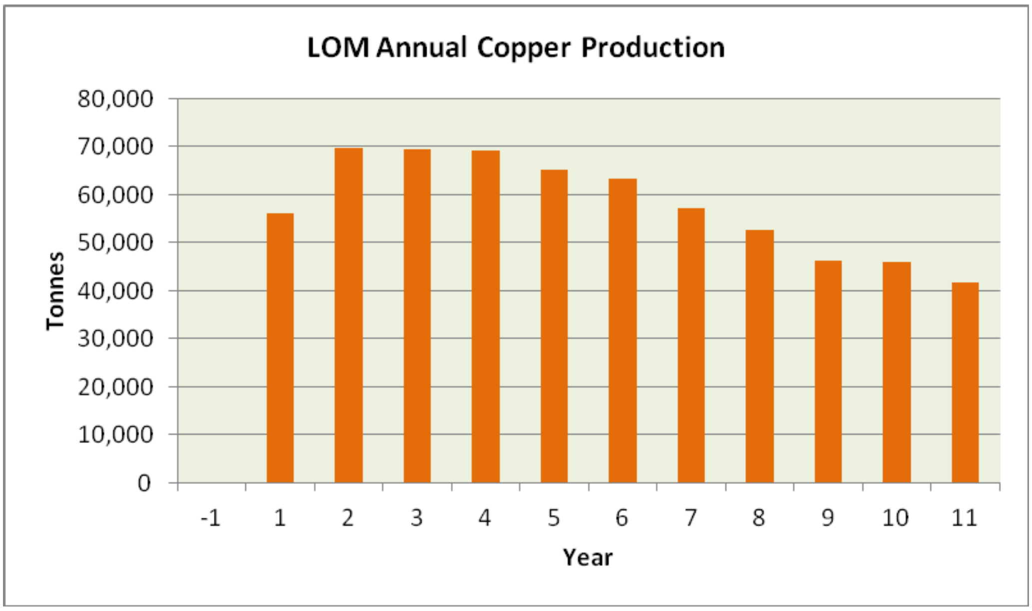
Ore mined and processed during the initial production years will be predominantly high oxide supergene material, and will contain only relatively minor quantities of primary sulphides. It will be required, therefore, to supplement the concentrate feed to the pressure oxidation leaching circuit with flotation concentrate high in primary sulphides, imported from a third party. Figure 24.3 illustrates the imported concentrate requirements on an annual basis for the LOM.

**Figure 24.3 LOM Imported Concentrate Requirements**




The LOM copper production, including the contained copper recovered from the imported flotation concentrate, is presented on an annual basis in Figure 24.4. The copper production peaks in Year 2 and 3 at circa 70 ktpa, primarily due to the higher copper head grade associated with the supergene material and also partly due to the copper contribution from the imported concentrate.

**Figure 24.4 LOM Copper Production**



## 25.0 INTERPRETATION AND CONCLUSIONS

Exploration, investigations and studies to date have identified that an opportunity exists for the development of a mine and processing plant at Kitumba, Zambia, to produce nominally 60,000 tpa of copper cathode for an eleven year mine life. Both the proposed mine and processing plant utilise techniques and technologies that are well established in the mining / processing industry.

The proposed operation will be based on exploitation of the Kitumba copper resource with modest quantities of copper concentrates purchased from other operations in Zambia to supplement the process plant feed in certain years.

Zambia is considered to be a mining friendly location with copper mining in particular well established, albeit not in the immediate vicinity of the proposed development.

Sufficient confidence in the outcomes of multi-disciplinary investigations and studies at a pre-feasibility study level of confidence exists to warrant more detailed investigations and the development of a definitive feasibility study to support a final investment decision.

## 26.0 RECOMMENDATIONS

Prior to embarking on project implementation additional site works, drilling and sampling, laboratory testwork and investigations and collection of data will have to be undertaken to fill gaps in the existing knowledge base.

Included in these activities are the following:

- Diamond drilling to obtain additional metallurgical samples.
- Hydrogeological work for mine design, quantification of mine dewatering requirements and identification of supplementary water resources.
- Preliminary geotechnical work on the proposed plant site and sites of major infrastructure items.
- Additional batch metallurgical variability testwork.
- Metallurgical pilot plant investigations.
- Additional mine geotechnical data collection and analysis.

The information generated from these activities will be combined with the output of existing and additional engineering and estimating work to complete a definitive feasibility study. This will complete the definition of project scope, provide ±10-15% capital and operating cost estimates for use as the basis for control budgets, and develop a preliminary execution plan and schedule for the project.

In conjunction with the OPFS consultants, Blackthorn has prepared and costed activities required to complete the definitive feasibility study and take the project through to an investment decision milestone. The activities and budget estimate are shown in Table 26.1.

**Table 26.1 Definitive Feasibility Study Budget Estimate**

| Activity | Total AUD |
|---|---|
| | **Revised Budget 30 May 2014** |
| Study management inc cost estimate and& report writing | 1,860,000 |
| Met testwork | 820,000 |
| Pilot Plant | 3,700,000 |
| Tailings inc waste rock and geochem | 346,000 |
| Hydrogeology and Geochemical (Exc drilling) | 683,144 |
| Environmental | 620,527 |
| Mine design and scheduling | 365,000 |
| Geotechnical Assessment | 619,000 |
| Drilling Costs (12,500m total) | 6,500,000 |
| BTR project management and corporate inc risk assessment | 775,000 |
| Peer reviews | 100,000 |
| Project expenses including 3rd party site visits | 120,000 |
| Marketing | 120,000 |
| Contingency (20%) | 2,815,279 |
| **Total For Final Investment Decision** | **19,443,950** |

Costs shown in Table 26.1 are in Australian Dollars and convert to approximately US$18.1 million at an exchange rate of AUD1.0 = US$0.93.

Some of the key cost elements, in particular the metallurgical testwork and pilot plant and additional drilling costs are well defined as a result of tender submissions having already been received or contracts already being in place with laboratories and drilling contractors, respectively.

It is recommended that work continues on activities related to the completion of a definitive study. In particular:

- Mining and geotechnical studies are completed to provide additional confidence in the mine design
- Testwork and pilot programs proceed to demonstrate proof of concept for the plant flowsheet
- Hydrogeology studies are completed to reduce uncertainty regarding water volumes expected to be encountered during mining and to firm up the inputs into the overall site water balance
- Discussions with ZESCO proceed with a view to signing a MoU for power supply for the future project

- Investigations and discussions are held with third parties within Zambia to increase confidence in the availability of key bulk reagents such as limestone and lime and terms for the purchase of copper concentrates to supplement feed to the POX autoclave.

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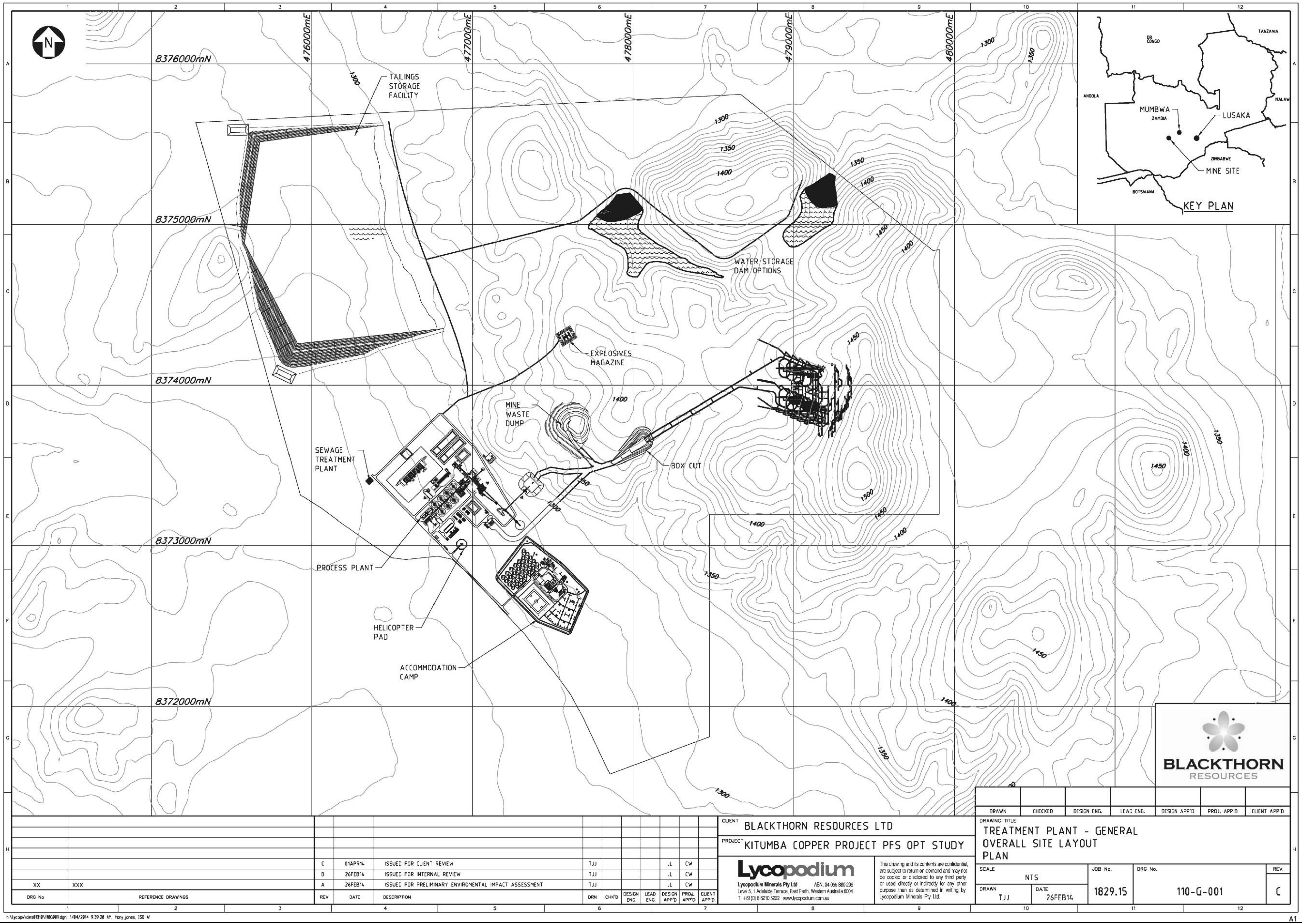
# APPENDIX 1

# PRELIMINARY OVERALL SITE LAYOUT


476000mE
477000mE
478000mE
479000mE
480000mE
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8375000mN
8374000mN
8373000mN
8372000mN
TAILINGS STORAGE FACILITY
WATER STORAGE DAM OPTIONS
EXPLOSIVES MAGAZINE
MINE WASTE DUMP
BOX CUT
SEWAGE TREATMENT PLANT
PROCESS PLANT
HELICOPTER PAD
ACCOMMODATION CAMP
KEY PLAN
TANZANIA
ANGOLA
MALAWI
MUMBWA
ZAMBIA
LUSAKA
ZIMBABWE
MINE SITE
BOTSWANA
BLACKTHORN RESOURCES
CLIENT BLACKTHORN RESOURCES LTD
PROJECT KITUMBA COPPER PROJECT PFS OPT STUDY
DRAWING TITLE TREATMENT PLANT - GENERAL OVERALL SITE LAYOUT PLAN
Lycopodium
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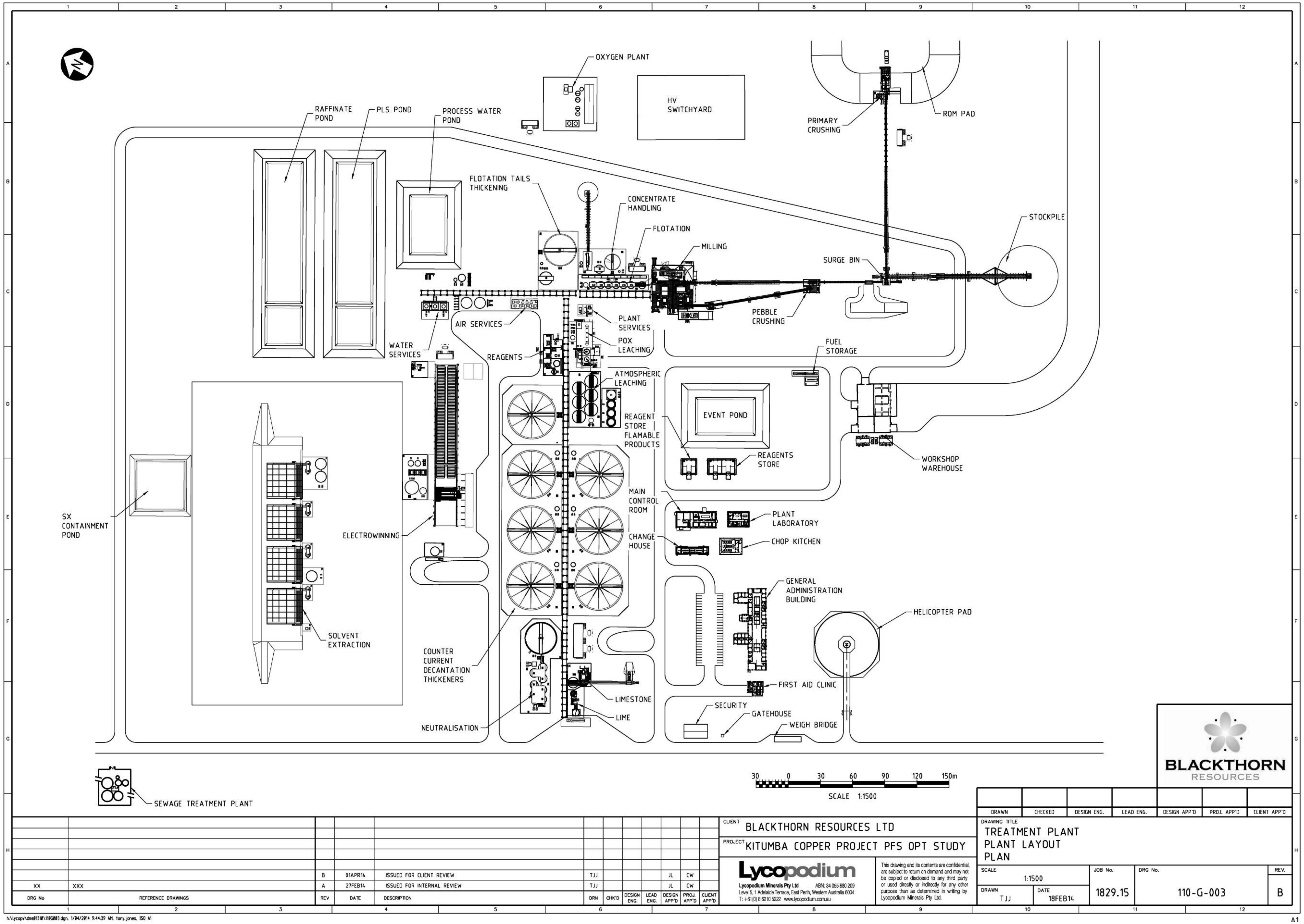
B 26FEB14 ISSUED FOR INTERNAL REVIEW
A 26FEB14 ISSUED FOR PRELIMINARY ENVIROMENTAL IMPACT ASSESSMENT
SCALE NTS
DRAWN TJJ
DATE 26FEB14
JOB No. 1829.15
DRG No. 110-G-001
REV. C
A1

# APPENDIX 2

# PRELIMINARY PLANT LAYOUT


OXYGEN PLANT
HV SWITCHYARD
RAFFINATE POND
PLS POND
PROCESS WATER POND
PRIMARY CRUSHING
ROM PAD
FLOTATION TAILS THICKENING
CONCENTRATE HANDLING
FLOTATION
MILLING
SURGE BIN
STOCKPILE
PEBBLE CRUSHING
AIR SERVICES
PLANT SERVICES
POX LEACHING
WATER SERVICES
REAGENTS
FUEL STORAGE
ATMOSPHERIC LEACHING
EVENT POND
REAGENT STORE FLAMABLE PRODUCTS
REAGENTS STORE
WORKSHOP WAREHOUSE
SX CONTAINMENT POND
ELECTROWINNING
MAIN CONTROL ROOM
PLANT LABORATORY
CHANGE HOUSE
CHOP KITCHEN
GENERAL ADMINISTRATION BUILDING
HELICOPTER PAD
SOLVENT EXTRACTION
COUNTER CURRENT DECANTATION THICKENERS
FIRST AID CLINIC
LIMESTONE
LIME
NEUTRALISATION
SECURITY
GATEHOUSE
WEIGH BRIDGE
SEWAGE TREATMENT PLANT
BLACKTHORN
RESOURCES
30 0 30 60 90 120 150m
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DRAWN CHECKED DESIGN ENG. LEAD ENG. DESIGN APP'D PROJ. APP'D CLIENT APP'D
CLIENT BLACKTHORN RESOURCES LTD
PROJECT KITUMBA COPPER PROJECT PFS OPT STUDY
DRAWING TITLE TREATMENT PLANT PLANT LAYOUT PLAN
Lycopodium
Lycopodium Minerals Pty Ltd ABN: 34 055 880 209
Level 5, 1 Adelaide Terrace, East Perth, Western Australia 6004
T: +61 (0) 8 6210 5222 www.lycopodium.com.au
This drawing and its contents are confidential, are subject to return on demand and may not be copied or disclosed to any third party or used directly or indirectly for any other purpose than as determined in writing by Lycopodium Minerals Pty Ltd.
SCALE 1:1500
DRAWN TJJ
DATE 18FEB14
JOB No. 1829.15
DRG No. 110-G-003
REV. B
B 01APR14 ISSUED FOR CLIENT REVIEW TJJ JL CW
A 27FEB14 ISSUED FOR INTERNAL REVIEW TJJ JL CW
REV DATE DESCRIPTION DRN CHK'D DESIGN ENG. LEAD ENG. DESIGN APP'D PROJ. APP'D CLIENT APP'D
XX XXX
DRG No REFERENCE DRAWINGS
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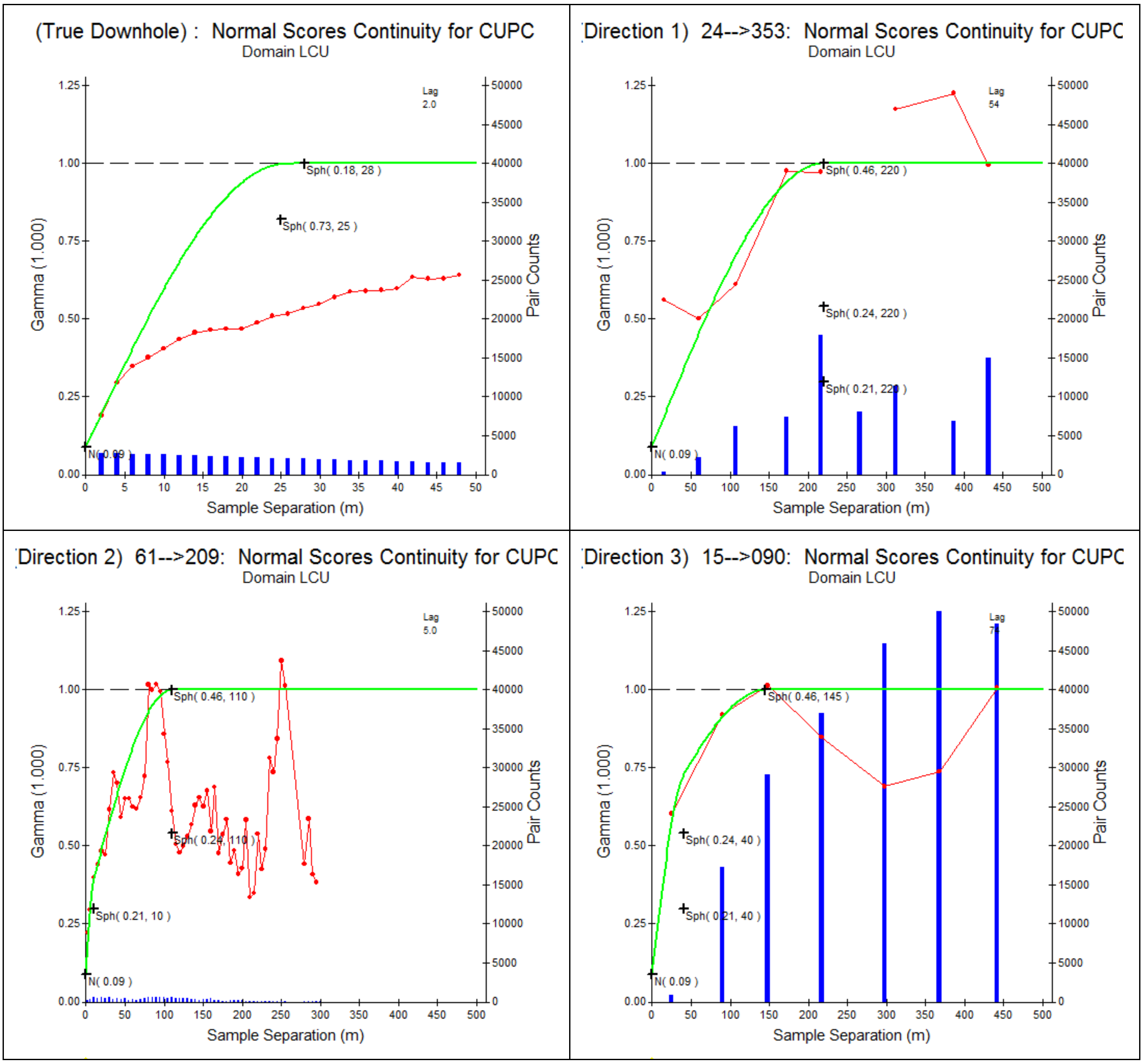
# APPENDIX 3

# VARIOGRAMS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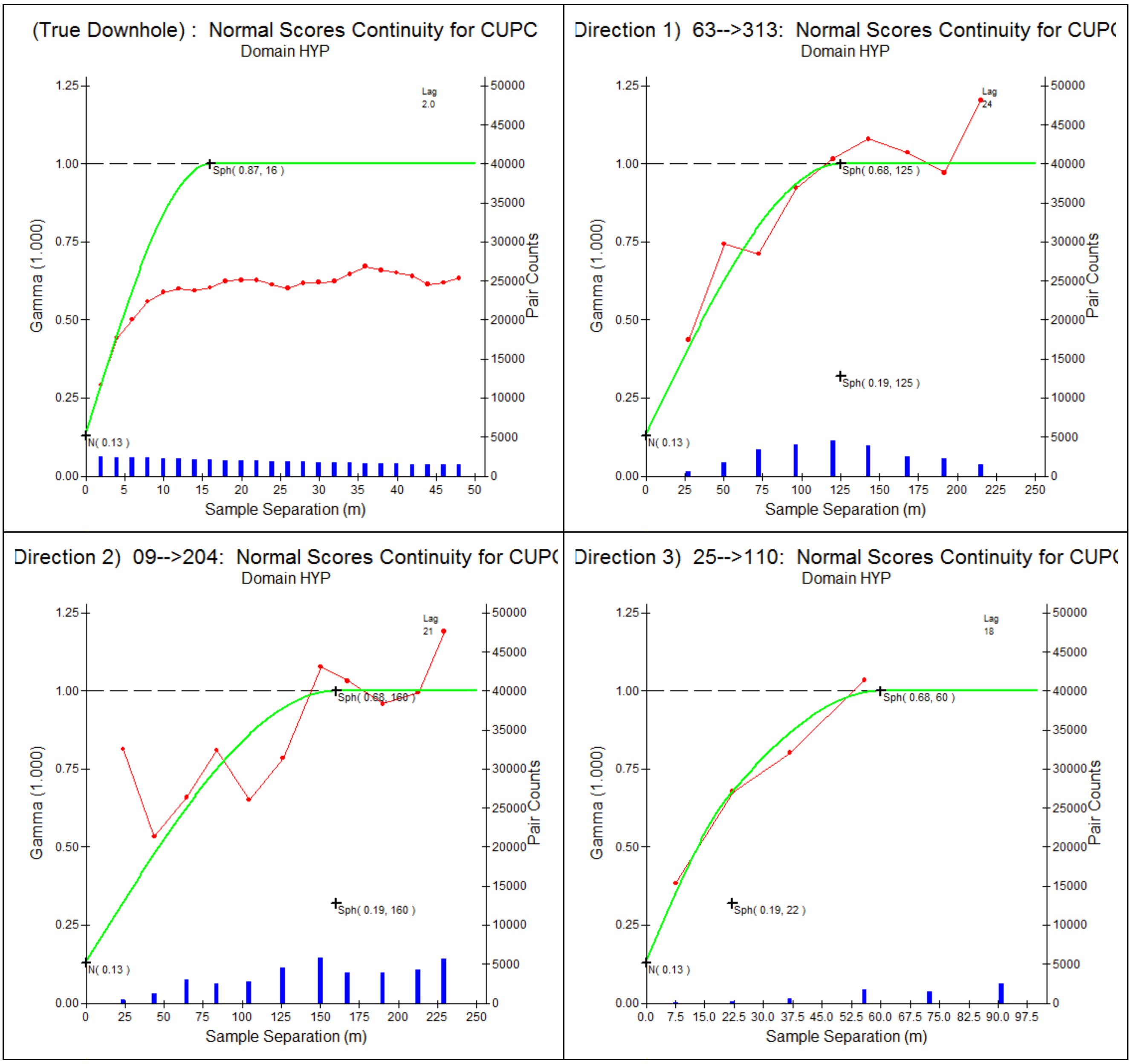
Domain LCU
Sph( 0.18, 28 )
Sph( 0.73, 25 )
N( 0.09 )
Lag 2.0
Gamma (1.000)
Pair Counts
Sample Separation (m)
Direction 1) 24-->353: Normal Scores Continuity for CUPC
Domain LCU
Sph( 0.46, 220 )
Sph( 0.24, 220 )
Sph( 0.21, 220 )
N( 0.09 )
Lag 54
Gamma (1.000)
Pair Counts
Sample Separation (m)
Direction 2) 61-->209: Normal Scores Continuity for CUPC
Domain LCU
Sph( 0.46, 110 )
Sph( 0.24, 110 )
Sph( 0.21, 10 )
N( 0.09 )
Lag 5.0
Gamma (1.000)
Pair Counts
Sample Separation (m)
Direction 3) 15-->090: Normal Scores Continuity for CUPC
Domain LCU
Sph( 0.46, 145 )
Sph( 0.24, 40 )
Sph( 0.21, 40 )
N( 0.09 )
Gamma (1.000)
Pair Counts
Sample Separation (m)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Domain HYP
Gamma (1.000)
Pair Counts
Sample Separation (m)
Lag 2.0
Sph( 0.87, 16 )
N( 0.13 )
Direction 1) 63-->313: Normal Scores Continuity for CUP(
Domain HYP
Gamma (1.000)
Pair Counts
Sample Separation (m)
Lag 24
Sph( 0.68, 125 )
Sph( 0.19, 125 )
N( 0.13 )
Direction 2) 09-->204: Normal Scores Continuity for CUP(
Domain HYP
Gamma (1.000)
Pair Counts
Sample Separation (m)
Lag 21
Sph( 0.68, 160 )
Sph( 0.19, 160 )
N( 0.13 )
Direction 3) 25-->110: Normal Scores Continuity for CUP(
Domain HYP
Gamma (1.000)
Pair Counts
Sample Separation (m)
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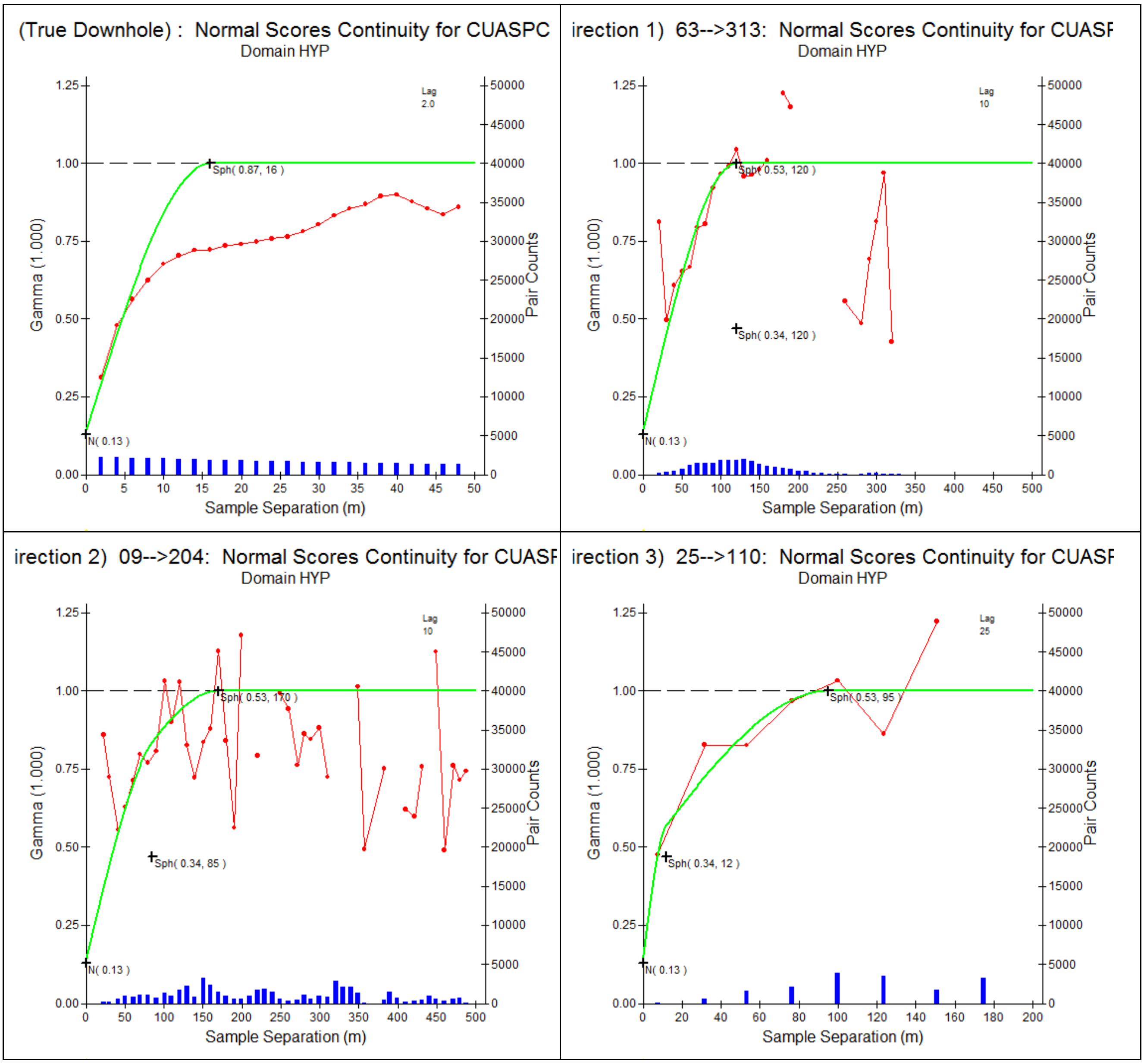
N( 0.13 )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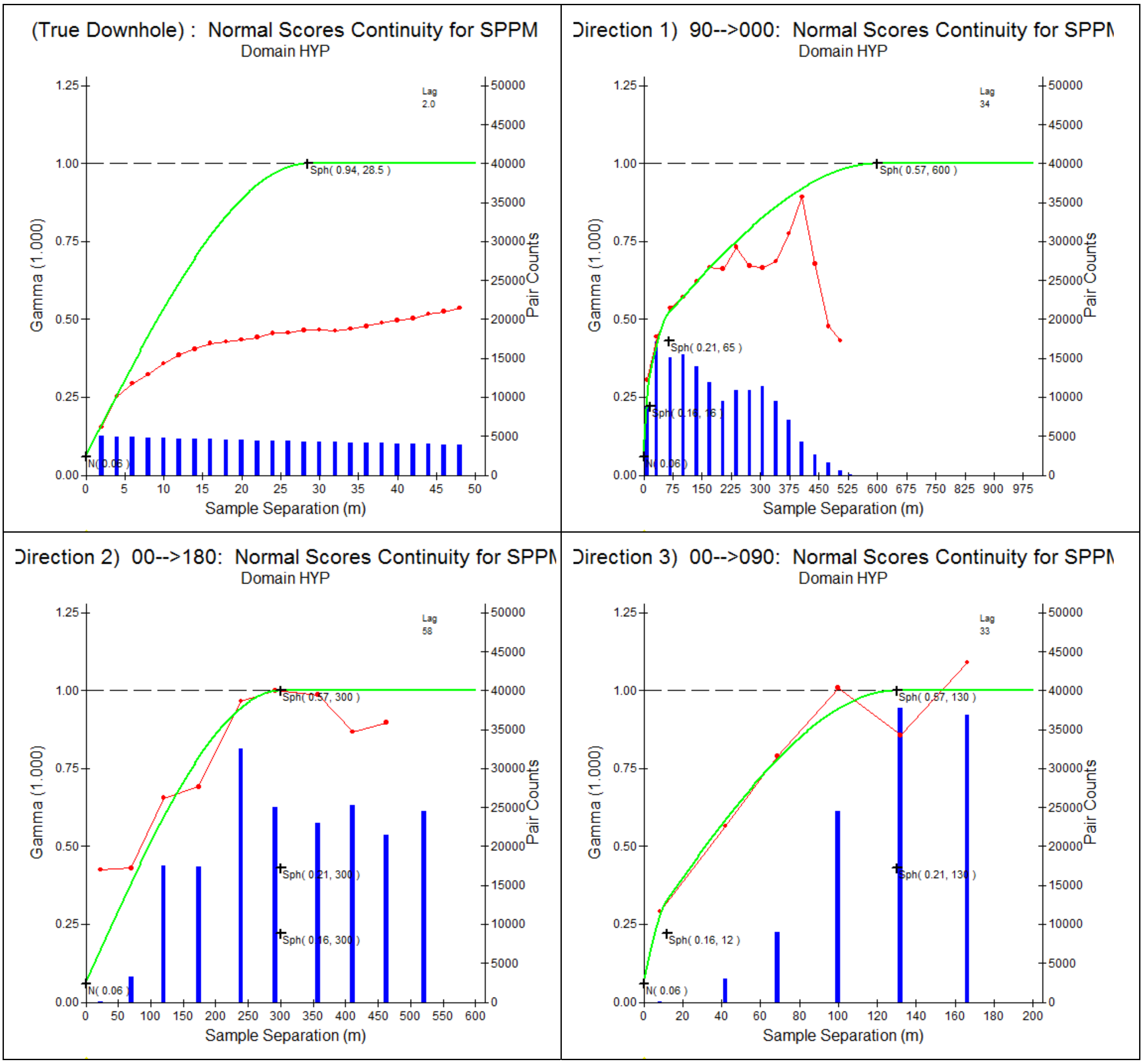
Domain HYP
Sph( 0.87, 16 )
N( 0.13 )
Gamma (1.000)
Pair Counts
Sample Separation (m)
irection 1) 63-->313: Normal Scores Continuity for CUASF
Domain HYP
Sph( 0.53, 120 )
Sph( 0.34, 120 )
N( 0.13 )
irection 2) 09-->204: Normal Scores Continuity for CUASF
Domain HYP
Sph( 0.53, 170 )
Sph( 0.34, 85 )
N( 0.13 )
irection 3) 25-->110: Normal Scores Continuity for CUASF
Domain HYP
Sph( 0.53, 95 )
Sph( 0.34, 12 )
N( 0.13 )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Domain HYP
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N( 0.06 )
Gamma (1.000)
Pair Counts
Sample Separation (m)
Direction 1) 90-->000: Normal Scores Continuity for SPPM
Domain HYP
Sph( 0.57, 600 )
Sph( 0.21, 65 )
Sph( 0.16, 16 )
N( 0.06 )
Direction 2) 00-->180: Normal Scores Continuity for SPPM
Domain HYP
Sph( 0.57, 300 )
Sph( 0.21, 300 )
Sph( 0.16, 300 )
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Domain HYP
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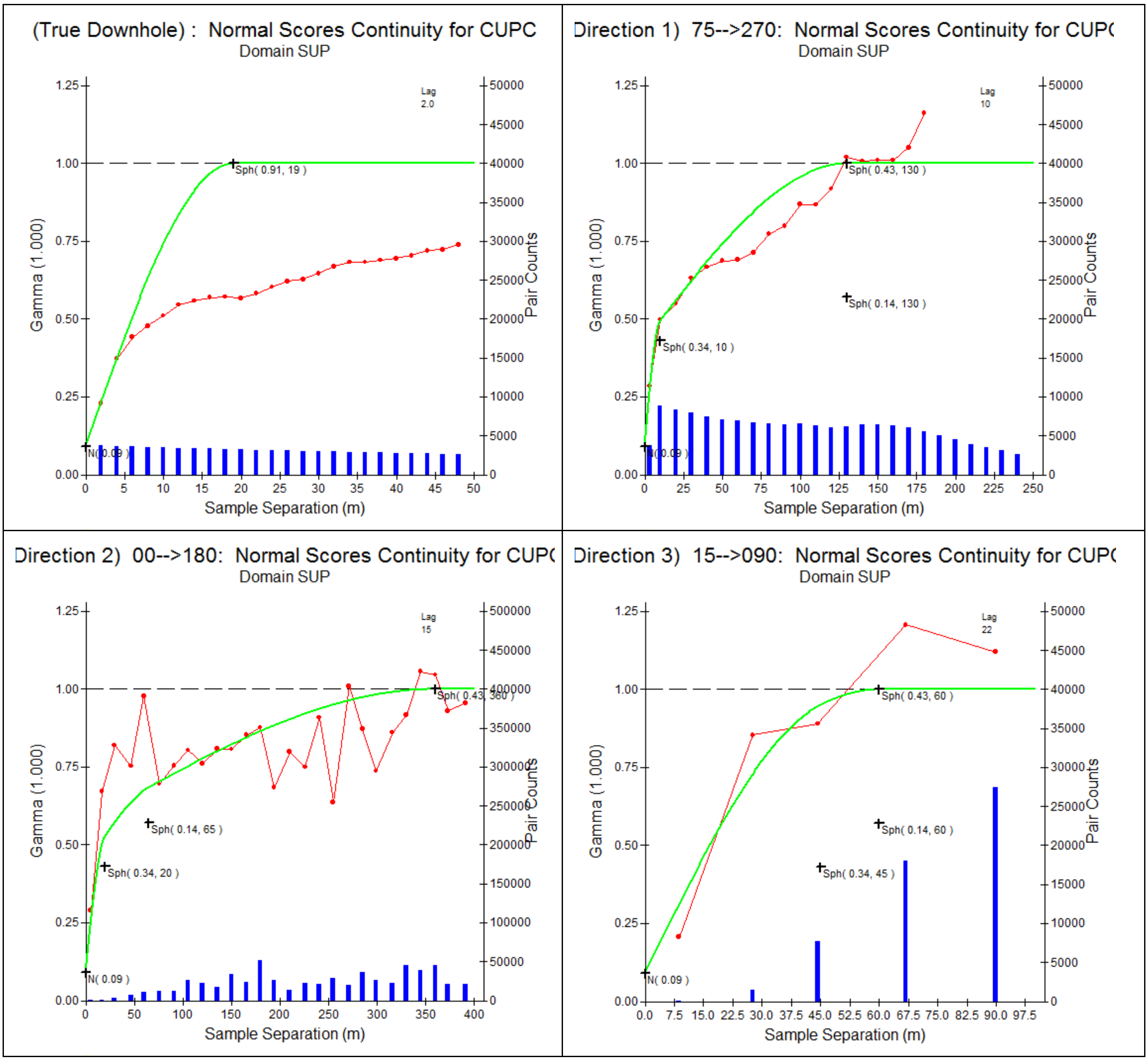
Sph( 0.21, 130 )
Sph( 0.16, 12 )


(True Downhole) : Normal Scores Continuity for CUPC
Domain SUP
Gamma (1.000)
Pair Counts
Sample Separation (m)
Lag 2.0
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N( 0.09 )
Direction 1) 75-->270: Normal Scores Continuity for CUP(
Domain SUP
Gamma (1.000)
Pair Counts
Sample Separation (m)
Lag 10
Sph( 0.43, 130 )
Sph( 0.14, 130 )
Sph( 0.34, 10 )
N( 0.09 )
Direction 2) 00-->180: Normal Scores Continuity for CUP(
Domain SUP
Gamma (1.000)
Pair Counts
Sample Separation (m)
Lag 15
Sph( 0.43, 360 )
Sph( 0.14, 65 )
Sph( 0.34, 20 )
N( 0.09 )
Direction 3) 15-->090: Normal Scores Continuity for CUP(
Domain SUP
Gamma (1.000)
Pair Counts
Sample Separation (m)
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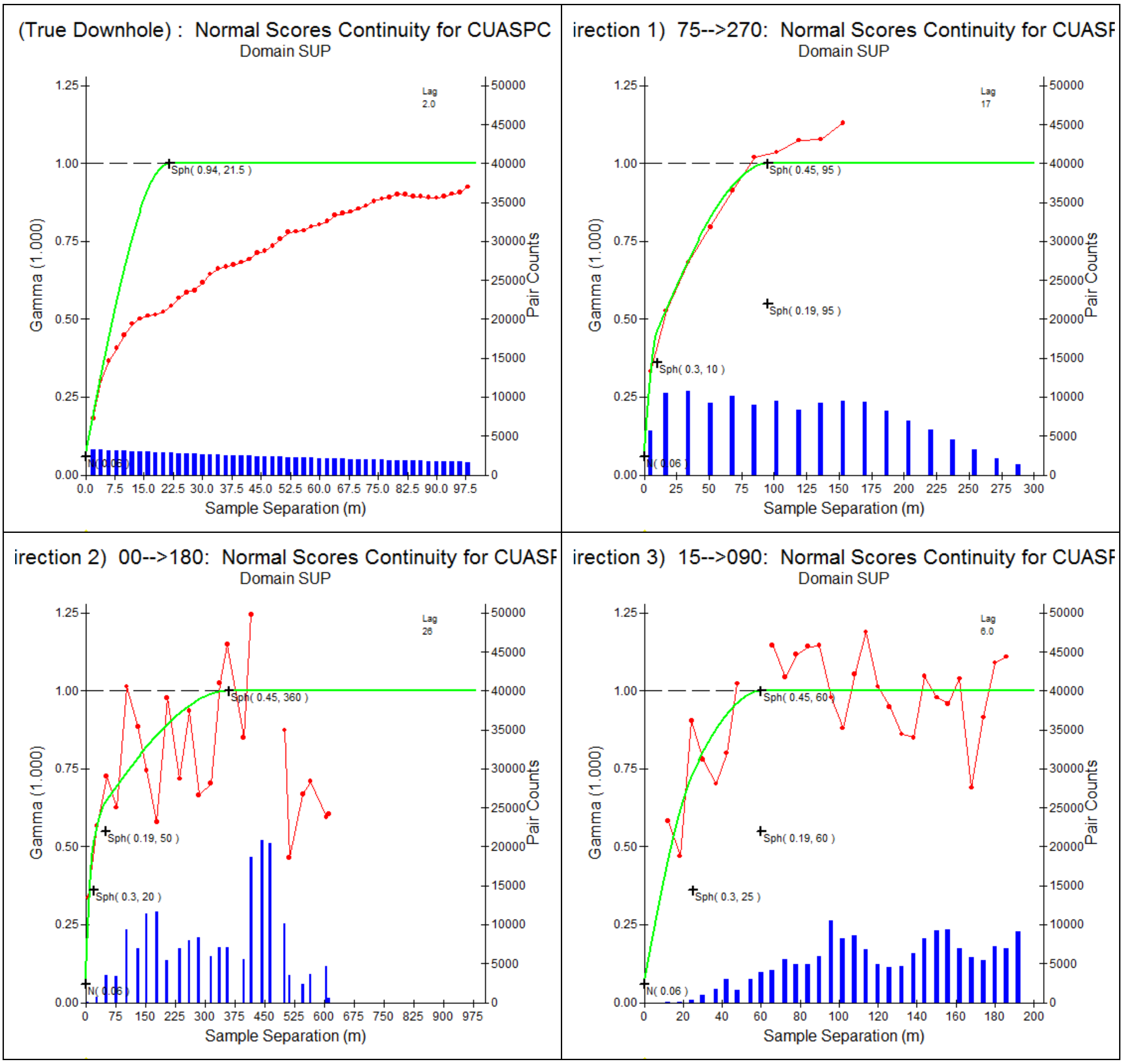
Sph( 0.14, 60 )
Sph( 0.34, 45 )
N( 0.09 )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Domain SUP
Gamma (1.000)
Pair Counts
Sample Separation (m)
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irection 1) 75-->270: Normal Scores Continuity for CUASF
Domain SUP
Sph( 0.45, 95 )
Sph( 0.19, 95 )
Sph( 0.3, 10 )
irection 2) 00-->180: Normal Scores Continuity for CUASF
Domain SUP
Sph( 0.45, 360 )
Sph( 0.19, 50 )
Sph( 0.3, 20 )
irection 3) 15-->090: Normal Scores Continuity for CUASF
Domain SUP
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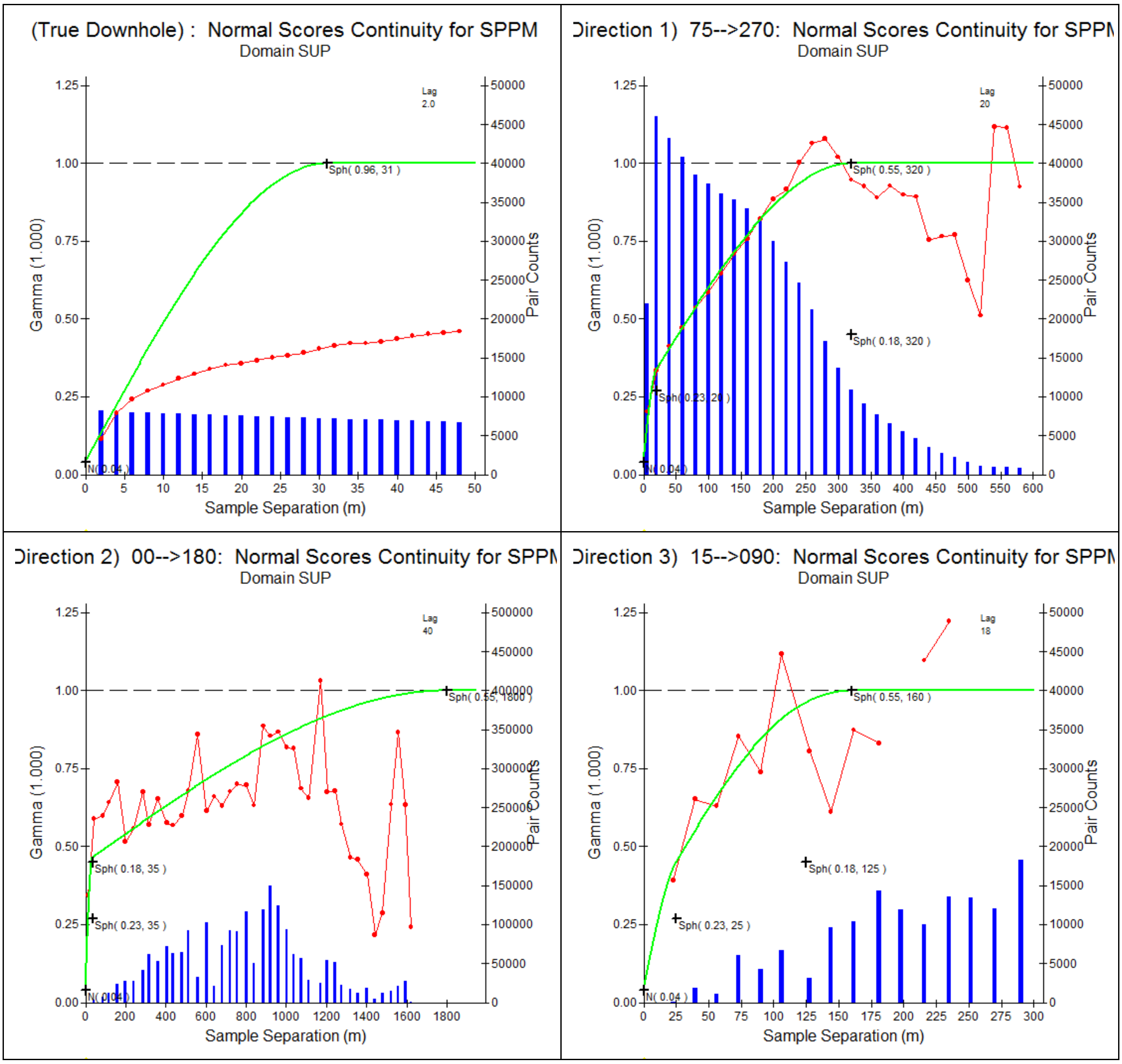
Sph( 0.19, 60 )
Sph( 0.3, 25 )
N( 0.06 )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Lag 2.0
Gamma (1.000)
Pair Counts
Sample Separation (m)
Direction 1) 75-->270: Normal Scores Continuity for SPPM
Domain SUP
Sph( 0.55, 320 )
Sph( 0.18, 320 )
Sph( 0.23, 20 )
N( 0.04 )
Lag 20
Gamma (1.000)
Pair Counts
Sample Separation (m)
Direction 2) 00-->180: Normal Scores Continuity for SPPM
Domain SUP
Sph( 0.55, 1800 )
Sph( 0.18, 35 )
Sph( 0.23, 35 )
N( 0.04 )
Lag 40
Gamma (1.000)
Pair Counts
Sample Separation (m)
Direction 3) 15-->090: Normal Scores Continuity for SPPM
Domain SUP
Sph( 0.55, 160 )
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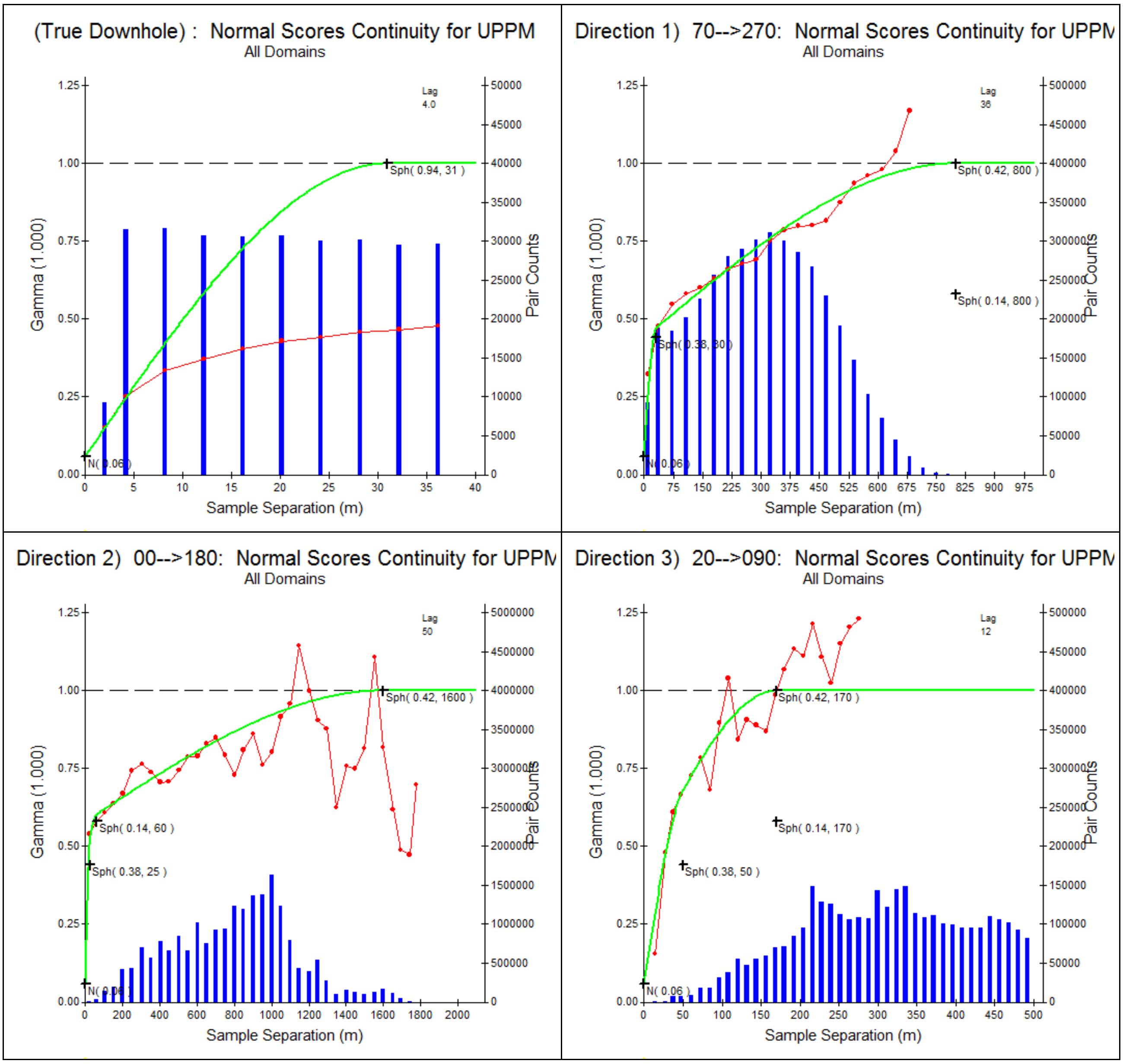
Sph( 0.23, 25 )
N( 0.04 )
Lag 18
Gamma (1.000)
Pair Counts
Sample Separation (m)


(True Downhole) : Normal Scores Continuity for UPPM
All Domains
Gamma (1.000)
Pair Counts
Sample Separation (m)
Lag 4.0
Sph( 0.94, 31 )
N( 0.06 )
Direction 1) 70-->270: Normal Scores Continuity for UPPM
All Domains
Lag 36
Sph( 0.42, 800 )
Sph( 0.14, 800 )
Sph( 0.38, 80 )
N( 0.06 )
Direction 2) 00-->180: Normal Scores Continuity for UPPM
All Domains
Lag 50
Sph( 0.42, 1600 )
Sph( 0.14, 60 )
Sph( 0.38, 25 )
N( 0.06 )
Direction 3) 20-->090: Normal Scores Continuity for UPPM
All Domains
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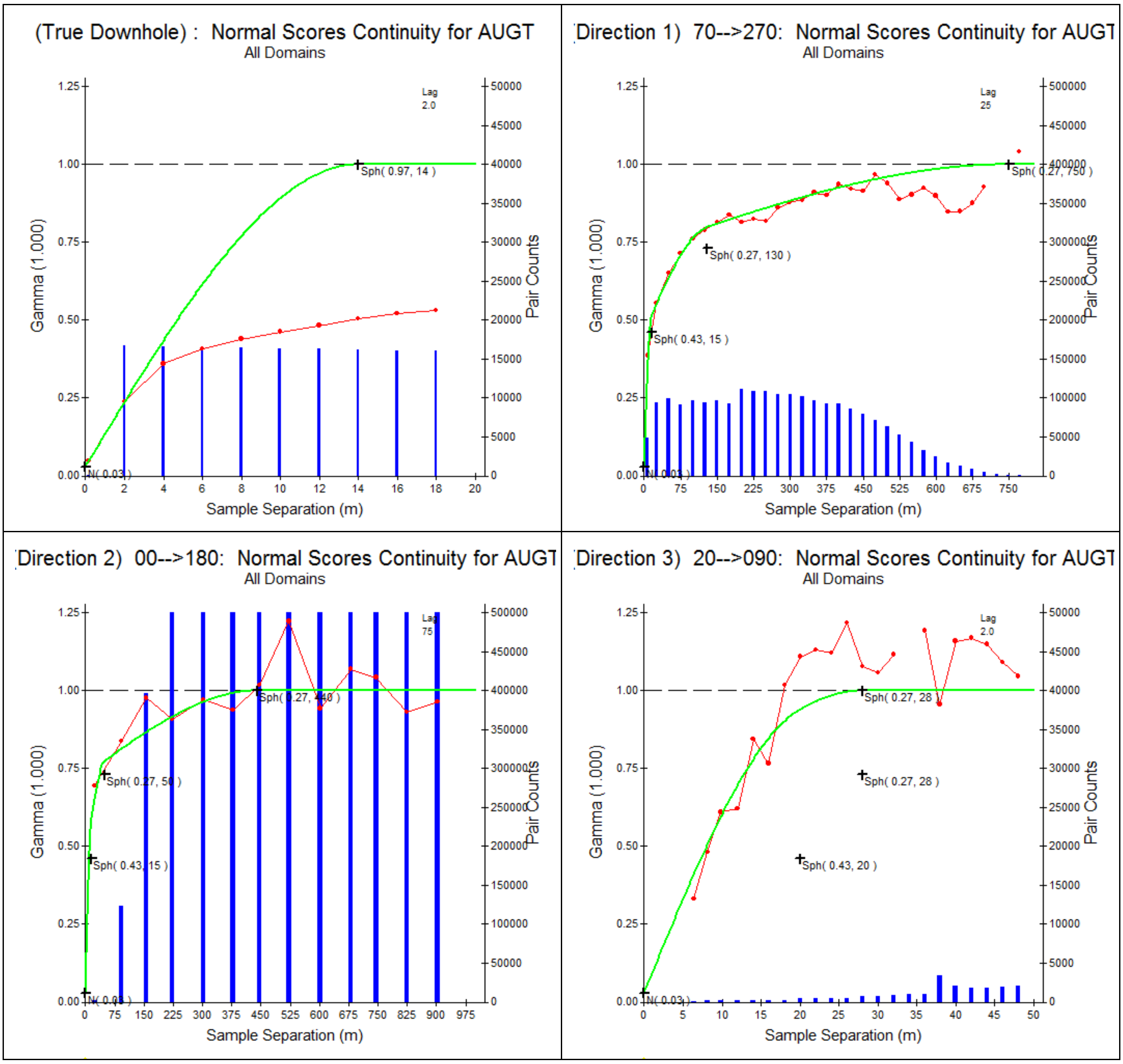
(True Downhole) : Normal Scores Continuity for AUGT
All Domains
Gamma (1.000)
Pair Counts
Sample Separation (m)
Lag 2.0
Sph( 0.97, 14 )
N( 0.03 )
Direction 1) 70-->270: Normal Scores Continuity for AUGT
All Domains
Gamma (1.000)
Pair Counts
Sample Separation (m)
Lag 25
Sph( 0.27, 750 )
Sph( 0.27, 130 )
Sph( 0.43, 15 )
N( 0.03 )
Direction 2) 00-->180: Normal Scores Continuity for AUGT
All Domains
Gamma (1.000)
Pair Counts
Sample Separation (m)
Lag 75
Sph( 0.27, 440 )
Sph( 0.27, 50 )
Sph( 0.43, 15 )
N( 0.03 )
Direction 3) 20-->090: Normal Scores Continuity for AUGT
All Domains
Gamma (1.000)
Pair Counts
Sample Separation (m)
Lag 2.0
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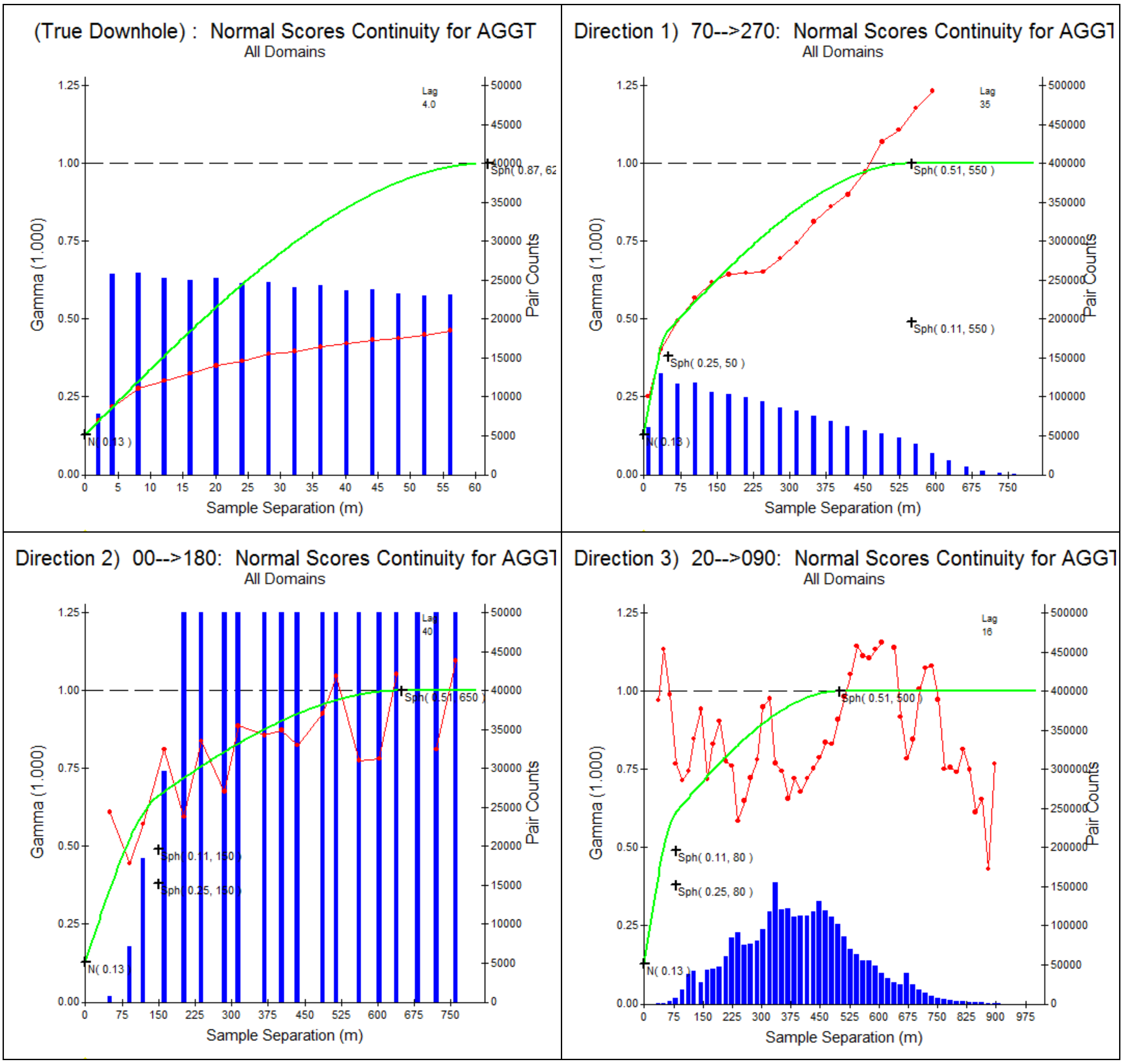
Sph( 0.27, 28 )
Sph( 0.43, 20 )
N( 0.03 )


(True Downhole) : Normal Scores Continuity for AGGT
All Domains
Gamma (1.000)
Pair Counts
Sample Separation (m)
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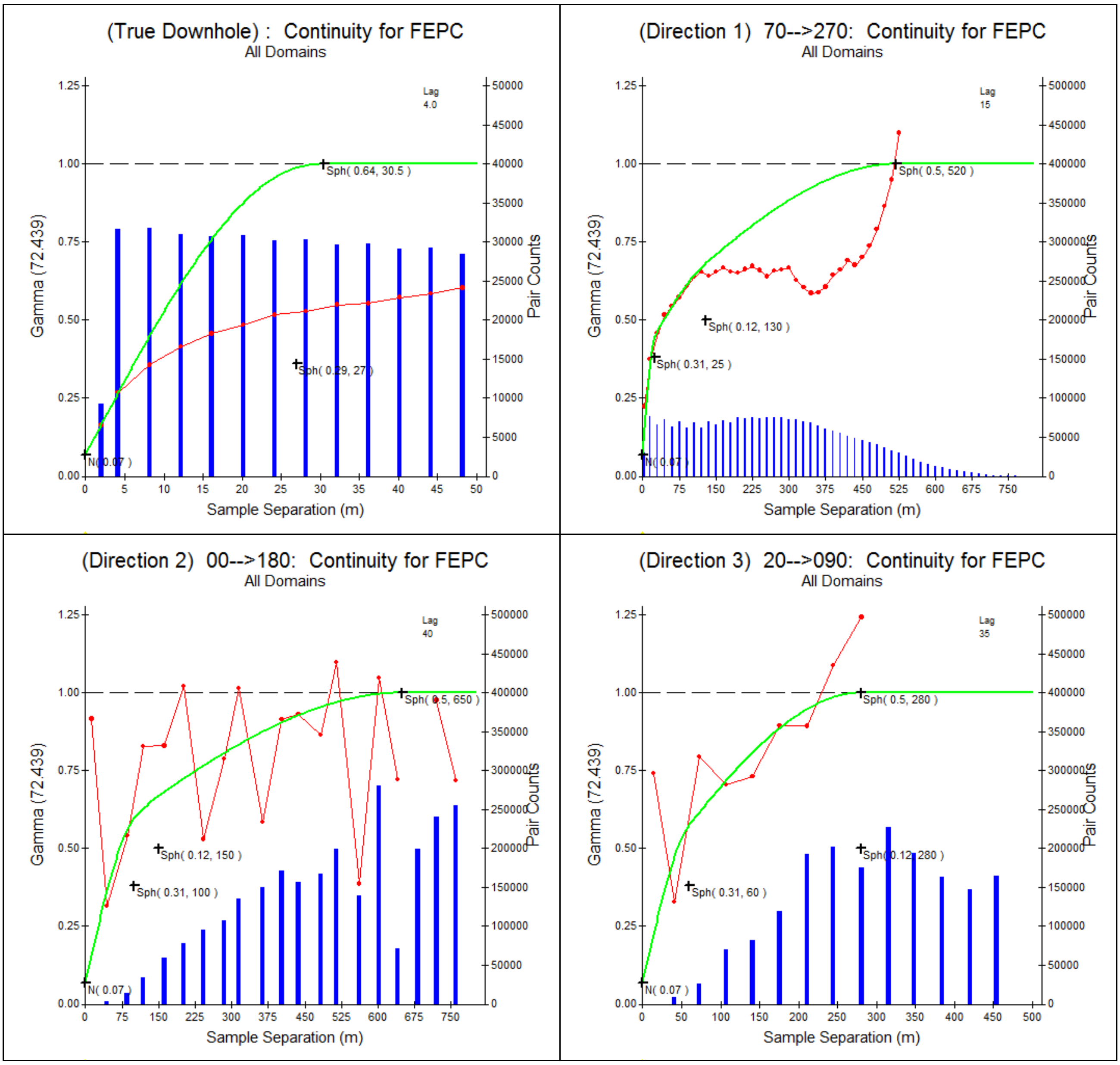
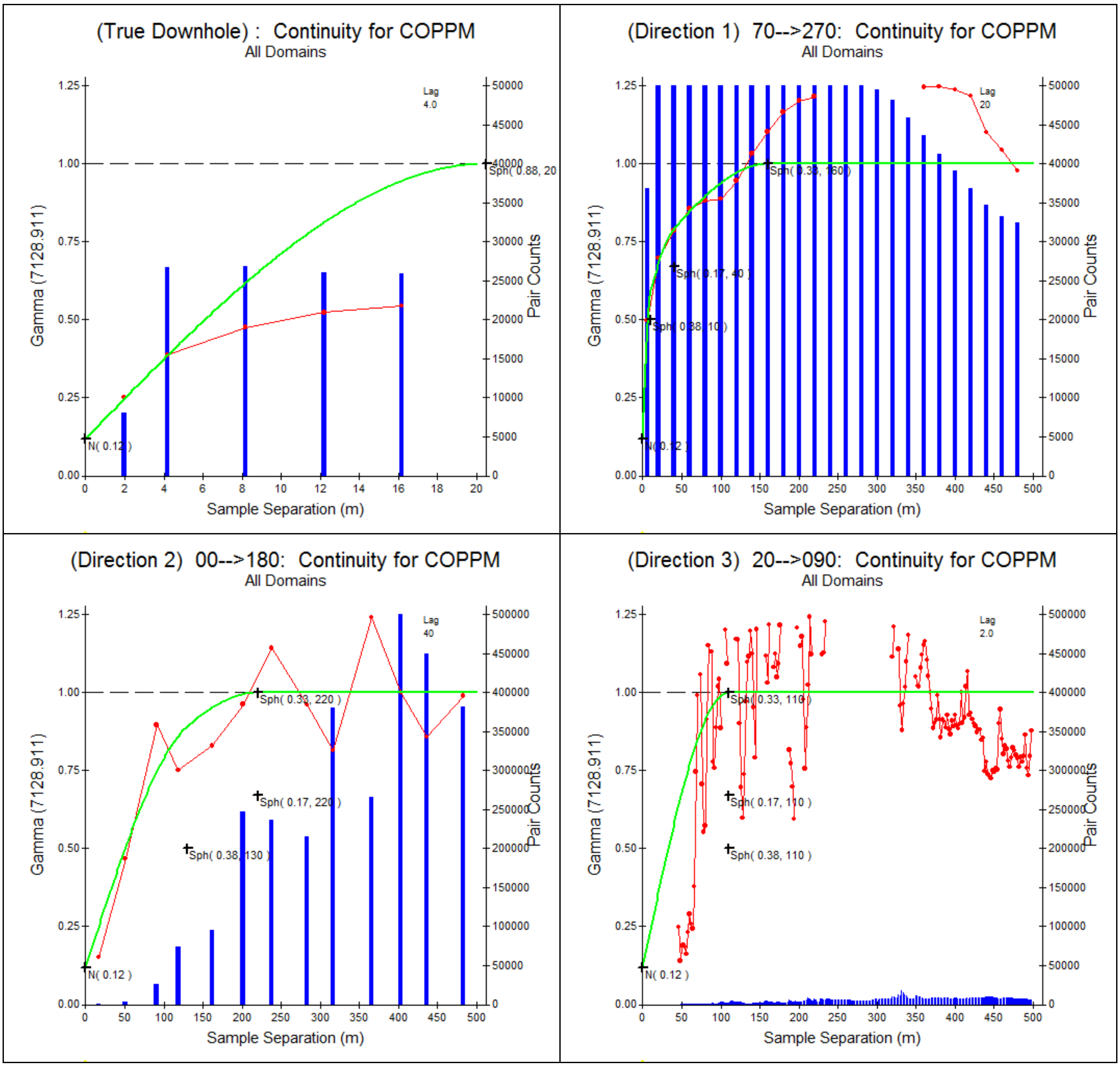
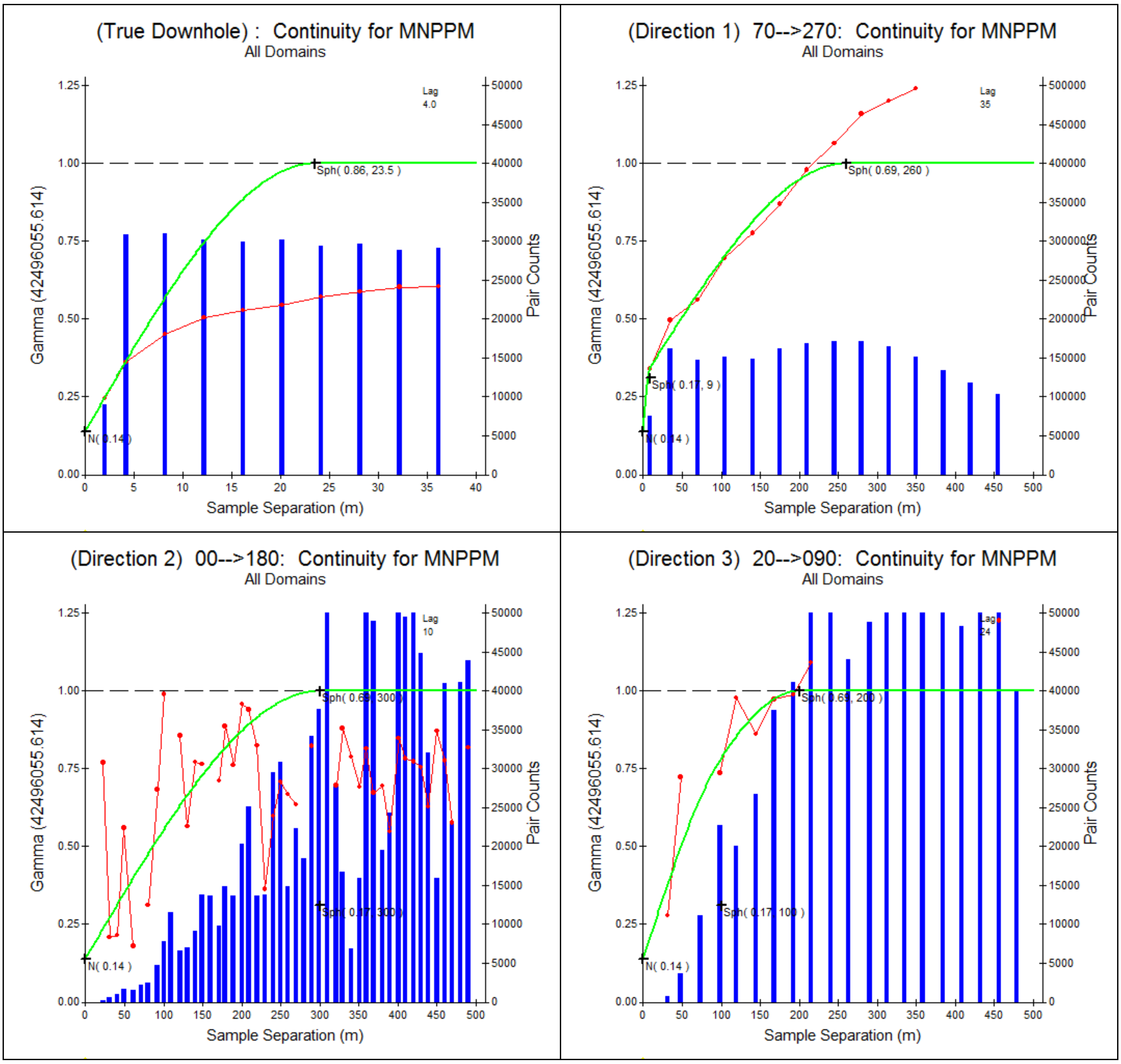
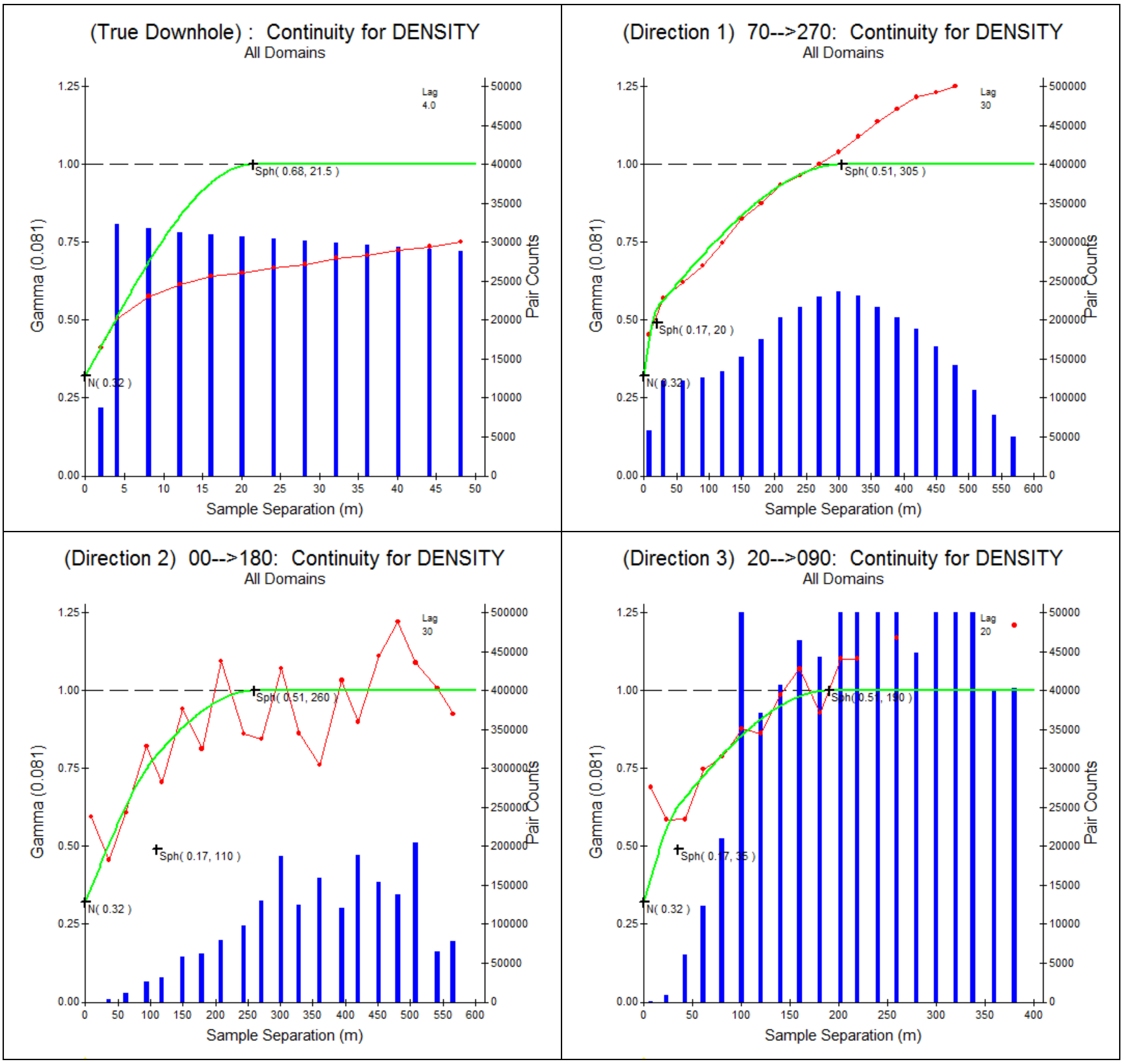
Sph( 0.87, 62
N( 0.13 )
Direction 1) 70-->270: Normal Scores Continuity for AGGT
All Domains
Gamma (1.000)
Pair Counts
Sample Separation (m)
Lag 35
Sph( 0.51, 550 )
Sph( 0.11, 550 )
Sph( 0.25, 50 )
N( 0.13 )
Direction 2) 00-->180: Normal Scores Continuity for AGGT
All Domains
Gamma (1.000)
Pair Counts
Sample Separation (m)
Lag 40
Sph( 0.51, 650 )
Sph( 0.11, 150 )
Sph( 0.25, 150 )
N( 0.13 )
Direction 3) 20-->090: Normal Scores Continuity for AGGT
All Domains
Gamma (1.000)
Pair Counts
Sample Separation (m)
Lag 16
Sph( 0.51, 500 )
Sph( 0.11, 80 )
Sph( 0.25, 80 )
N( 0.13 )


(True Downhole) : Continuity for FEPC
All Domains
Lag
4.0
Sph( 0.64, 30.5 )
Sph( 0.29, 27 )
N( 0.07 )
Gamma (72.439)
Pair Counts
Sample Separation (m)
(Direction 1) 70-->270: Continuity for FEPC
All Domains
Lag
15
Sph( 0.5, 520 )
Sph( 0.12, 130 )
Sph( 0.31, 25 )
N( 0.07 )
Gamma (72.439)
Pair Counts
Sample Separation (m)
(Direction 2) 00-->180: Continuity for FEPC
All Domains
Lag
40
Sph( 0.5, 650 )
Sph( 0.12, 150 )
Sph( 0.31, 100 )
N( 0.07 )
Gamma (72.439)
Pair Counts
Sample Separation (m)
(Direction 3) 20-->090: Continuity for FEPC
All Domains
Lag
35
Sph( 0.5, 280 )
Sph( 0.12, 280 )
Sph( 0.31, 60 )
N( 0.07 )
Gamma (72.439)
Pair Counts
Sample Separation (m)

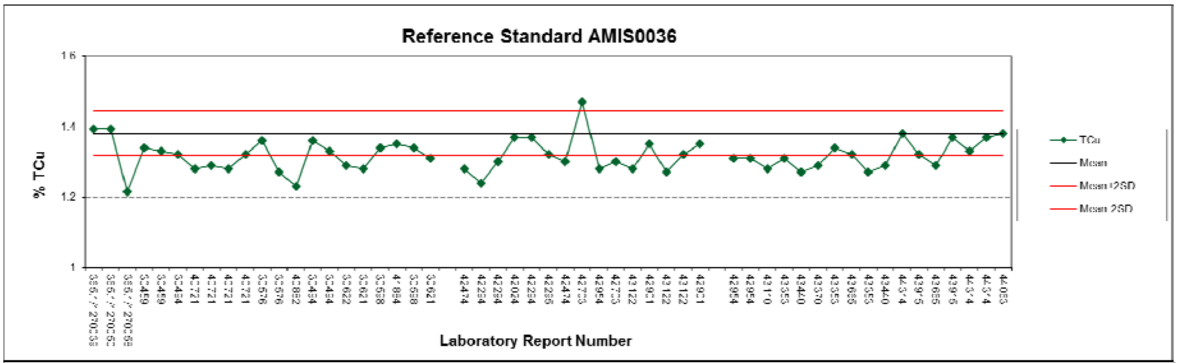

(True Downhole) : Continuity for COPPM
All Domains
Gamma (7128.911)
Pair Counts
Sample Separation (m)
Lag 4.0
N( 0.12 )
Sph( 0.88, 20
(Direction 1) 70-->270: Continuity for COPPM
All Domains
Gamma (7128.911)
Pair Counts
Sample Separation (m)
Lag 20
N( 0.12 )
Sph( 0.38, 10 )
Sph( 0.17, 40 )
Sph( 0.33, 160 )
(Direction 2) 00-->180: Continuity for COPPM
All Domains
Gamma (7128.911)
Pair Counts
Sample Separation (m)
Lag 40
N( 0.12 )
Sph( 0.38, 130 )
Sph( 0.17, 220 )
Sph( 0.33, 220 )
(Direction 3) 20-->090: Continuity for COPPM
All Domains
Gamma (7128.911)
Pair Counts
Sample Separation (m)
Lag 2.0
N( 0.12 )
Sph( 0.38, 110 )
Sph( 0.17, 110 )
Sph( 0.33, 110 )

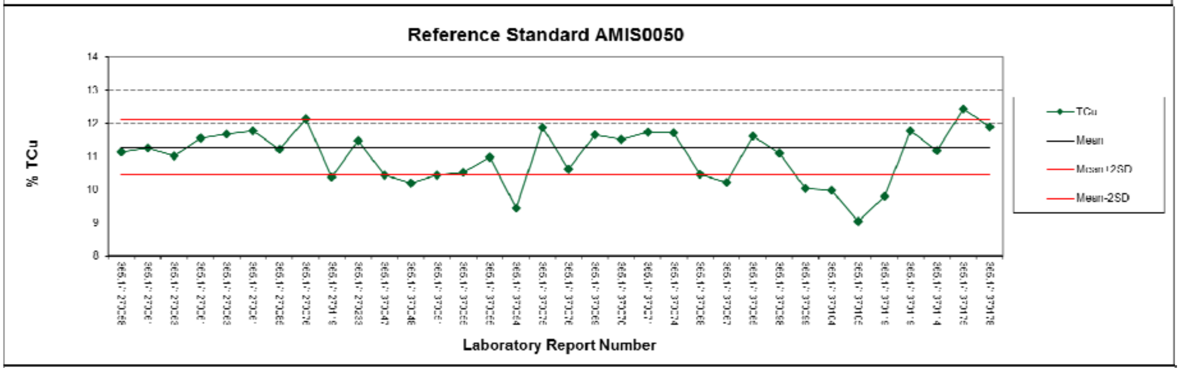

(True Downhole) : Continuity for MNPPM
All Domains
Gamma (42496055.614)
Pair Counts
Sample Separation (m)
Lag 4.0
Sph( 0.86, 23.5 )
N( 0.14 )
(Direction 1) 70-->270: Continuity for MNPPM
All Domains
Gamma (42496055.614)
Pair Counts
Sample Separation (m)
Lag 35
Sph( 0.69, 260 )
Sph( 0.17, 9 )
N( 0.14 )
(Direction 2) 00-->180: Continuity for MNPPM
All Domains
Gamma (42496055.614)
Pair Counts
Sample Separation (m)
Lag 10
Sph( 0.69, 300 )
Sph( 0.17, 300 )
N( 0.14 )
(Direction 3) 20-->090: Continuity for MNPPM
All Domains
Gamma (42496055.614)
Pair Counts
Sample Separation (m)
Lag 24
Sph( 0.69, 200 )
Sph( 0.17, 100 )
N( 0.14 )

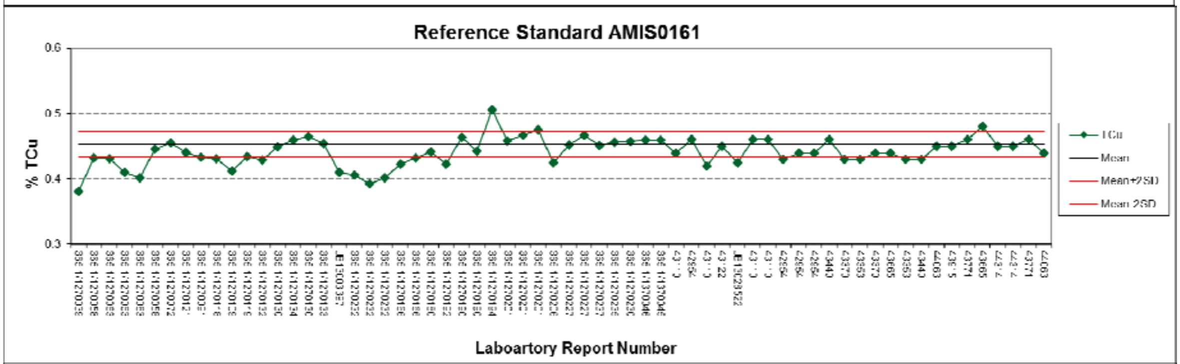

(True Downhole) : Continuity for DENSITY
All Domains
Gamma (0.081)
Pair Counts
Sample Separation (m)
Lag 4.0
Sph( 0.68, 21.5 )
N( 0.32 )
(Direction 1) 70-->270: Continuity for DENSITY
All Domains
Gamma (0.081)
Pair Counts
Sample Separation (m)
Lag 30
Sph( 0.51, 305 )
Sph( 0.17, 20 )
N( 0.32 )
(Direction 2) 00-->180: Continuity for DENSITY
All Domains
Gamma (0.081)
Pair Counts
Sample Separation (m)
Lag 30
Sph( 0.51, 260 )
Sph( 0.17, 110 )
N( 0.32 )
(Direction 3) 20-->090: Continuity for DENSITY
All Domains
Gamma (0.081)
Pair Counts
Sample Separation (m)
Lag 20
Sph( 0.51, 190 )
Sph( 0.17, 35 )
N( 0.32 )

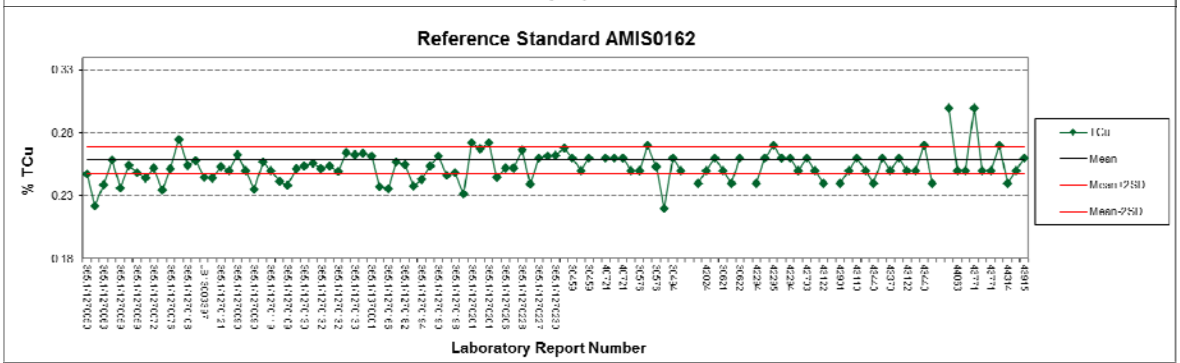
# APPENDIX 4

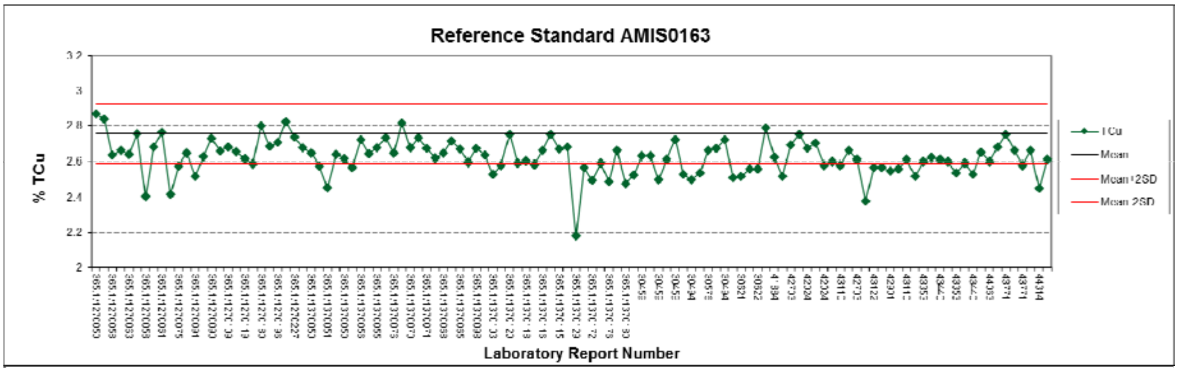
# QUALITY CONTROL CHARTS

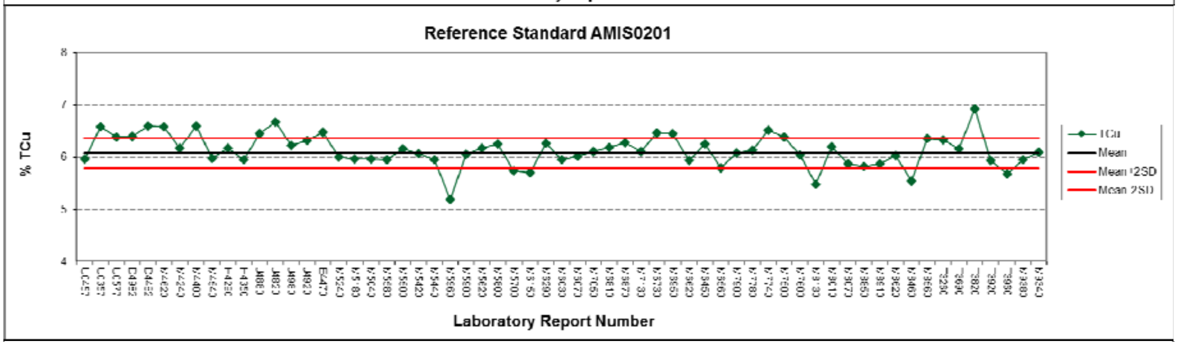

Reference Standard AMIS0036
% TCu
1.6
1.4
1.2
1
Laboratory Report Number
TCu
Mean
Mean+2SD
Mean-2SD

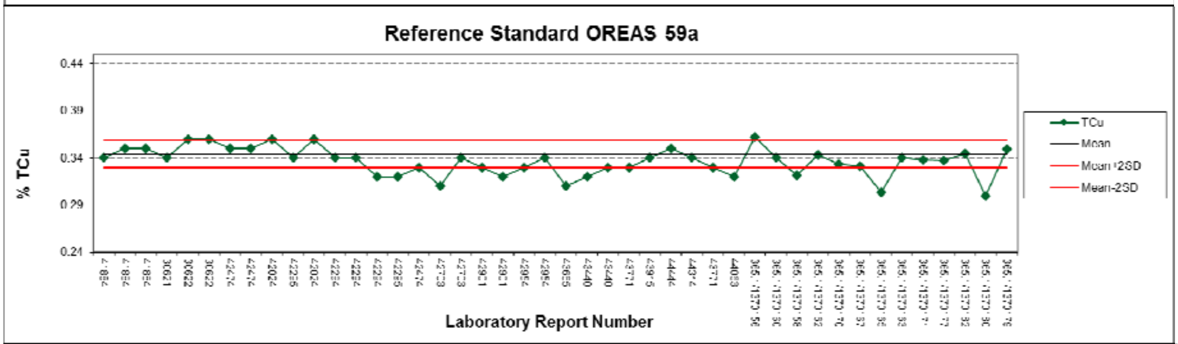

Reference Standard AMIS0050
% TCu
14
13
12
11
10
9
8
Laboratory Report Number
TCu
Mean
Mean+2SD
Mean-2SD

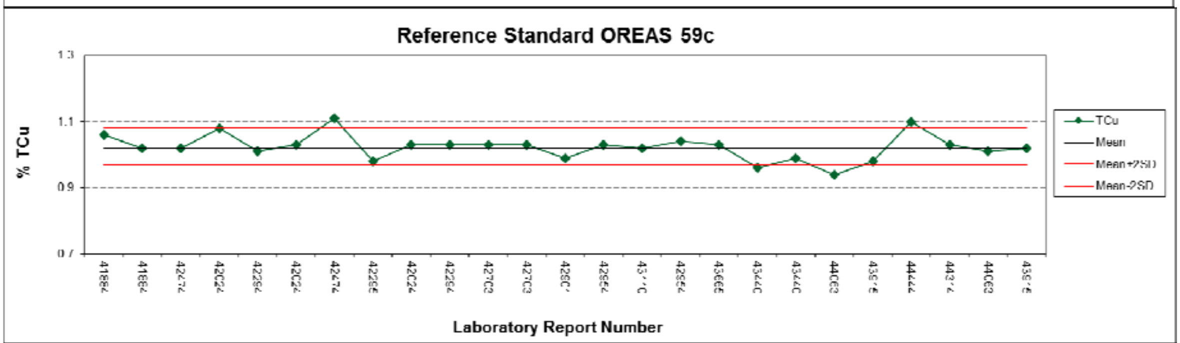

Reference Standard AMIS0161
% TCu
0.6
0.5
0.4
0.3
Laboartory Report Number
TCu
Mean
Mean+2SD
Mean-2SD

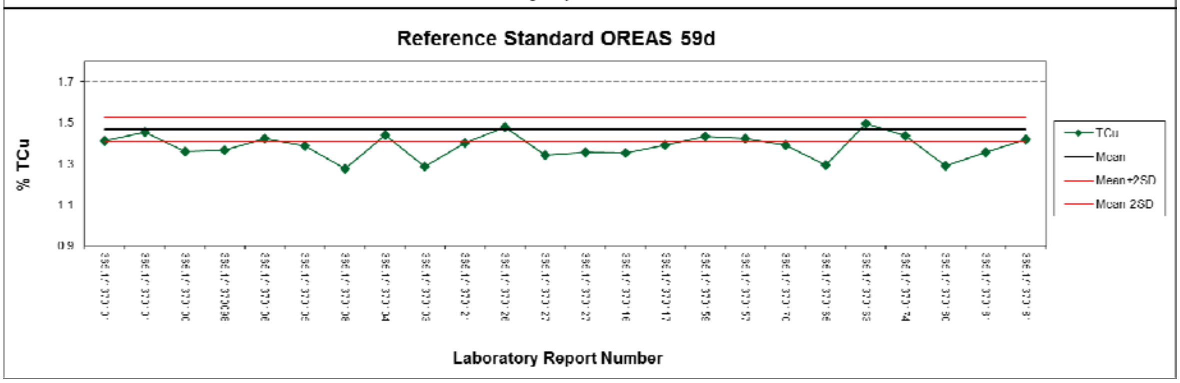

Reference Standard AMIS0162
% TCu
Laboratory Report Number
TCu
Mean
Mean+2SD
Mean-2SD

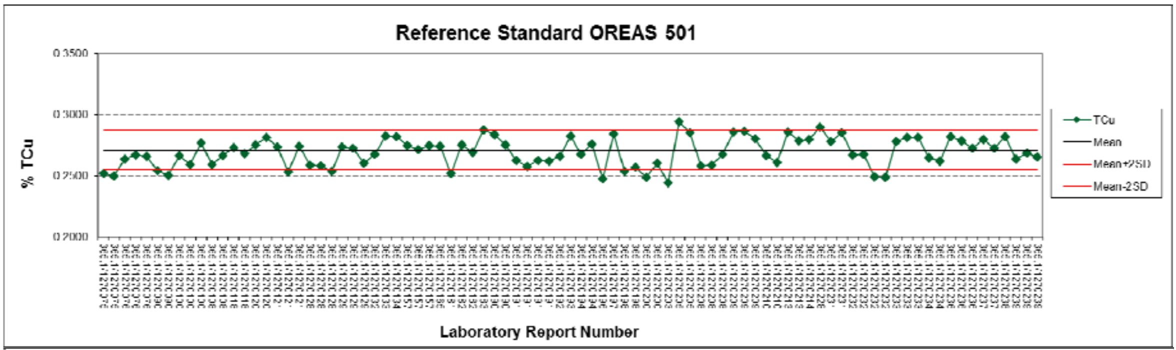

Reference Standard AMIS0163
% TCu
Laboratory Report Number
TCu
Mean
Mean+2SD
Mean-2SD

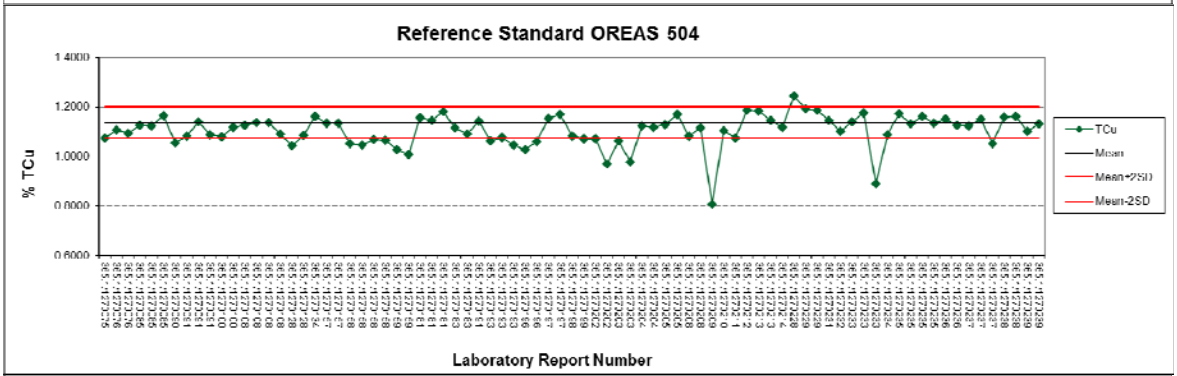

Reference Standard AMIS0201
% TCu
Laboratory Report Number
TCu
Mean
Mean+2SD
Mean-2SD

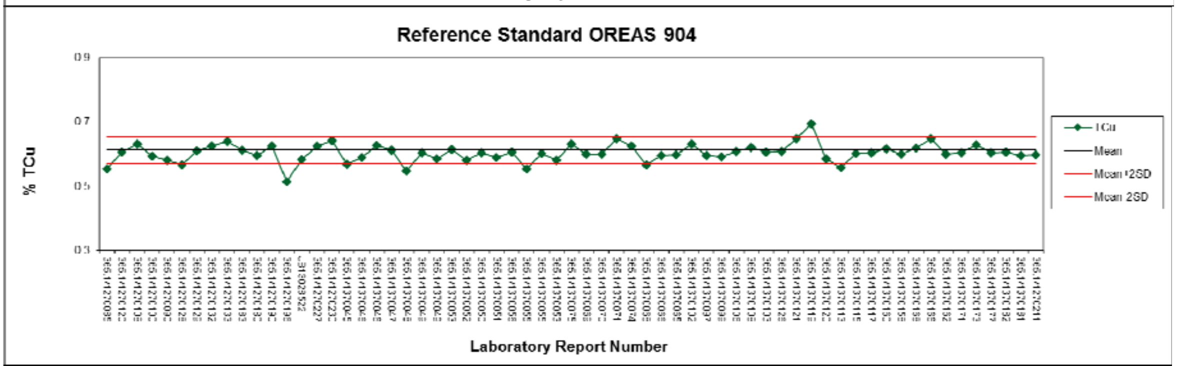

Reference Standard OREAS 59a
% TCu
Laboratory Report Number
TCu
Mean
Mean+2SD
Mean-2SD

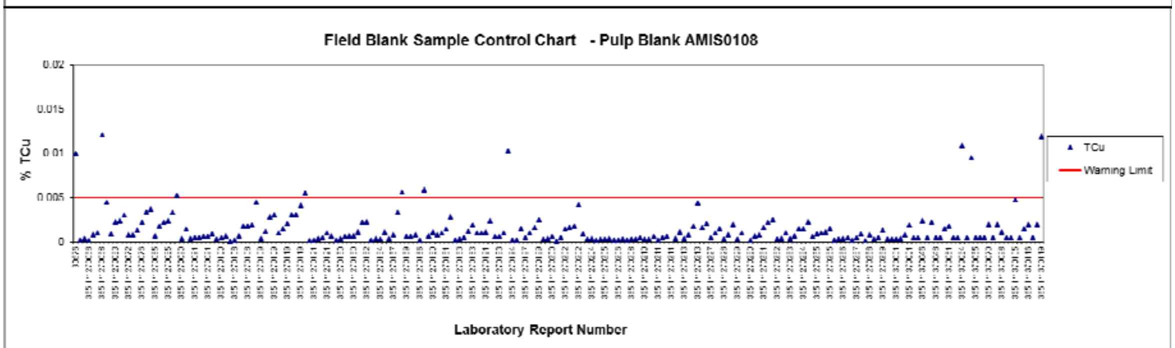

Reference Standard OREAS 59c
% TCu
Laboratory Report Number
TCu
Mean
Mean+2SD
Mean-2SD

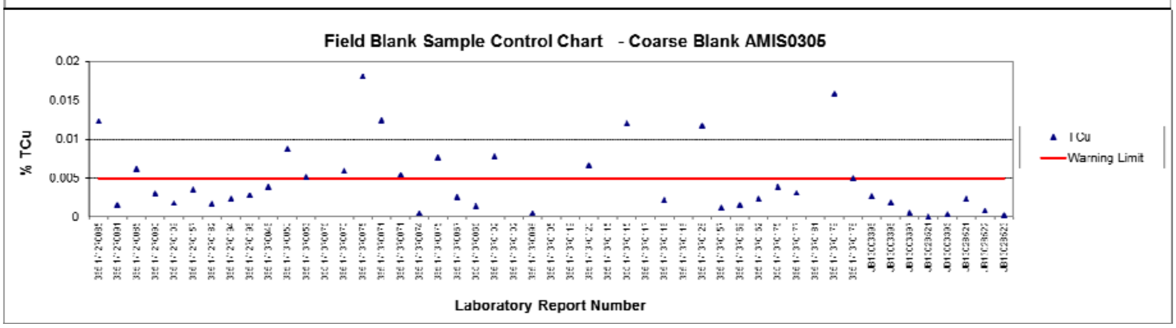

Reference Standard OREAS 59d
% TCu
Laboratory Report Number
TCu
Mean
Mean+2SD
Mean-2SD

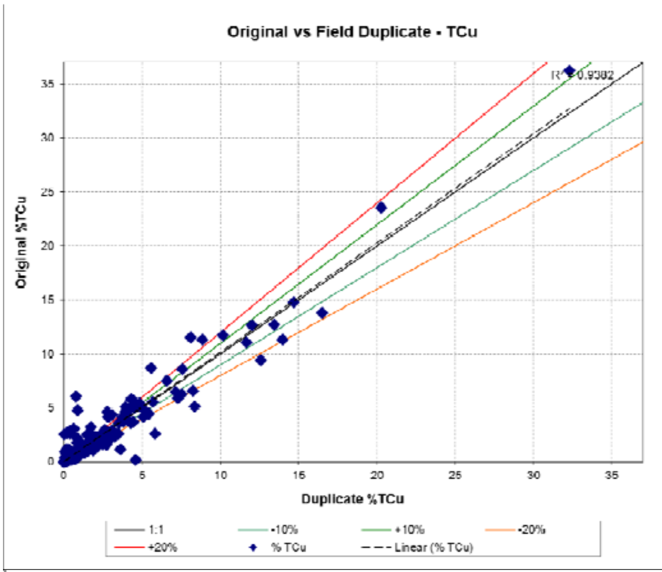

Reference Standard OREAS 501
% TCu
Laboratory Report Number
TCu
Mean
Mean+2SD
Mean-2SD

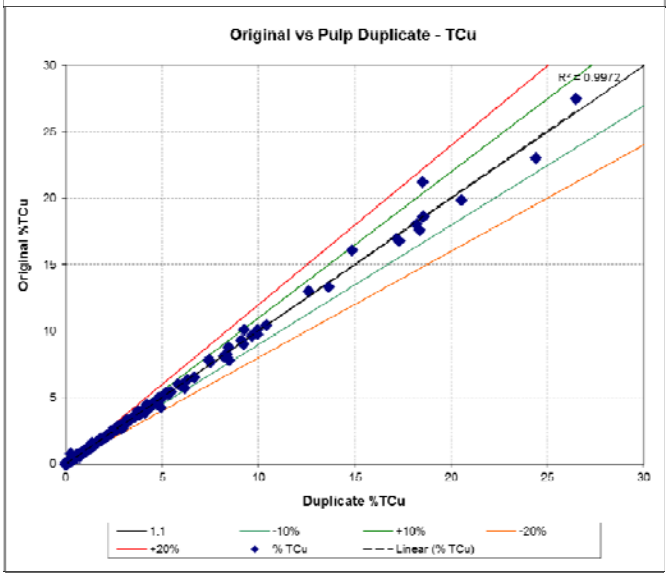

Reference Standard OREAS 504
% TCu
Laboratory Report Number
TCu
Mean
Mean+2SD
Mean-2SD

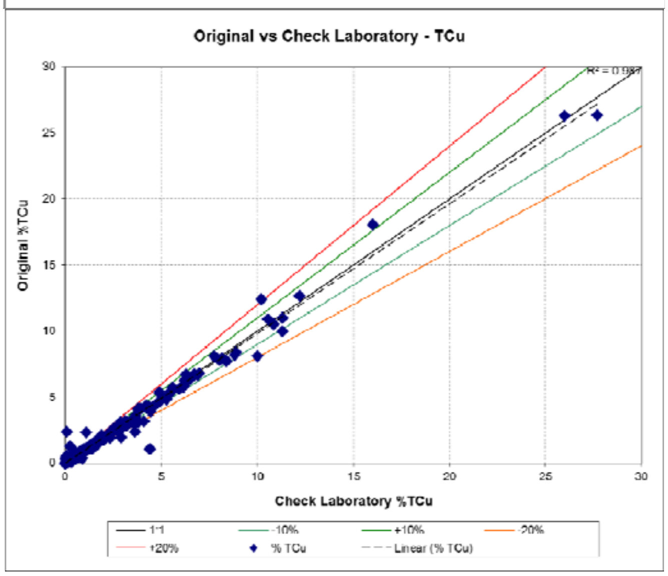

Reference Standard OREAS 904
% TCu
Laboratory Report Number


Field Blank Sample Control Chart - Pulp Blank AMIS0108
% TCu
Laboratory Report Number
TCu
Warning Limit


Field Blank Sample Control Chart - Coarse Blank AMIS0305
% TCu
Laboratory Report Number
Warning Limit


Original vs Field Duplicate - TCu
Original %TCu
Duplicate %TCu
1:1
-10%
+10%
-20%
+20%
% TCu
Linear (% TCu)


Original vs Pulp Duplicate - TCu
Original %TCu
Duplicate %TCu
1:1
-10%
+10%
-20%
+20%
% TCu
Linear (% TCu)


Original vs Check Laboratory - TCu
Original %TCu
Check Laboratory %TCu
1:1
-10%
+10%
-20%
+20%
% TCu
Linear (% TCu)

# APPENDIX 5

# QUALIFIED PERSONS CERTIFICATES

## CERTIFICATE OF QUALIFIED PERSON

I, Christopher Gorden Waller, MAusIMM(CP), as an author of this report entitled Kitumba Copper Project Optimised Pre-feasibility Study NI 43-101 Technical Report (Revision D) for the Kitumba Copper Project, Zambia, prepared for Intrepid Mines Limited and dated 25th September 2014, do hereby certify that:

1) I am Manager of Studies with Lycopodium Minerals Pty Ltd. My office address is Level 5, 1 Adelaide Terrace, East Perth, Western Australia 6004.

2) I am a graduate of the South Australian Institute of Technology (now The University of South Australia) 1978 with a Bachelor of Applied Science degree in Applied Chemistry.

3) I am a Member of the Australasian Institute of Mining and Metallurgy, membership number 101899 and registered as a Chartered Professional with that Institute. I have worked as a metallurgist, operations manager and study manager for a total of thirty six years since my graduation. My relevant experience for the purpose of the Technical Report is:

   - review and report as a consultant on numerous process facilities and mining projects around the world for due diligence and regulatory requirements

   - Study Manager on a number of feasibility studies in the mining industry in Africa, Australia and Asia.

   - operational roles to the level of General Manager at a number of mining operations in Africa and Australia (including Zambia).

4) I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

5) I have not visited the Kitumba Copper Project site.

6) I am responsible for all of preparation of Item Numbers 1 (sub-sections: 1, 2, 9, 13, 14, 15, 16, 17, 18, 19, 20), 2, 3, 4, 5 (except 5.5), 6 (jointly with MSA), 13, 17, 18 (except 18.1.4, 18.1.5, 18.2, 18.3), 19, 20, 21, 22, 23, 24, 25, 26, 27 (jointly) of the Technical Report.

7) I am independent of the Issuer applying the test set out in Section 1.5.(4) of NI 43-101.

8) I have / have not been involved in any previous Technical Report on the Kitumba Copper Project.

9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10) 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 6th day of October 2014

Christopher Gorden Waller, MAusIMM(CP)

**CERTIFICATE OF QUALIFIED PERSON**

I, Michael James Robertson, Pr.Sci.Nat., as an author of this report entitled Kitumba Copper Project Optimised Pre-feasibility Study NI 43-101 Technical Report (Revision D) for the Kitumba Copper Project, Zambia, prepared for Intrepid Mines Limited and dated 25th September 2014, do hereby certify that:

1) I am a Principal Consulting Geologist with The MSA Group Pty Ltd. My office address is 20B Rothesay Avenue, Craighall Park, Johannesburg, South Africa 2196.

2) I am a graduate of the University of the Witwatersrand in 1985 with a Bachelor of Science in Engineering degree in Mining Geology.

3) I am a Member of the Geological Society of South Africa and registered as a Professional Natural Scientist with the South African Council for Natural Scientific Professions. I have worked as a geologist for a total of 23 years since my graduation. My relevant experience for the purpose of the Technical Report is:

   - Planning, execution and management of base metal and gold exploration projects in Africa and the Middle East

   - Mineral property assessments, exploration programme audits, and scoping to feasibility study inputs on a number of projects in Africa, the Middle East and Asia

   - review and reporting as a consultant on numerous exploration and mining projects for regulatory requirements

4) I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

5) I have visited the Kitumba Copper Project on numerous occasions since 2006, the most recent visit being from 21 to 25 July 2014.

6) I am responsible for all of preparation of Item Numbers: 1 (sub-sections: 3, 4, 5, 6, 7, 8), 6 (jointly with Lycopodium), 7, 8, 9, 10, 11, 12 and 27 (jointly) of the Technical Report.

7) I am independent of the Issuer applying the test set out in Section 1.5.(4) of NI 43-101.

8) I have not been involved in any previous NI43-101 Technical Report on the Kitumba Copper Project.

9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10) 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 6th October 2014

Michael James Robertson, Pr,Sci.Nat. (Professional Natural Scientist)

## CERTIFICATE OF QUALIFIED PERSON

I, Jeremy Charles Witley, Pr.Sci.Nat., as an author of this report entitled Kitumba Copper Project Optimised Pre-feasibility Study NI 43-101 Technical Report (Revision D) for the Kitumba Copper Project, Zambia, prepared for Intrepid Mines Limited and dated 25th September 2014, do hereby certify that:

1) I am a Principal Resource Consultant with The MSA Group Pty Ltd. My office address is 20B Rothesay Avenue, Craighall Park, Johannesburg, South Africa 2196.

2) I am a graduate of the University of Leicester in 1988 with a Bachelor of Science (Hons) degree in Mining Geology.

3) I am a Member of the Geological Society of South Africa and registered as a Professional Natural Scientist with the South African Council for Natural Scientific Professions. I have worked as a geologist for a total of 26 years since my graduation. My relevant experience for the purpose of the Technical Report is:

   - Varied experience in a number of Central African Copper belt deposits as a consultant with the MSA Group and Snowden Mining Industry Consultants

   - 10 years of experience at nickel-copper mines in north east Botswana

   - Considerable experience in Mineral Resource estimation for a variety of base and precious metal deposits.

4) I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

5) I visited the Kitumba Copper Project in November 2012 for four days and again for four days in August 2013.

6) I am responsible for Item Numbers 1 (sub-section 10) and 14 of the Technical Report.

7) I am independent of the Issuer applying the test set out in Section 1.5.(4) of NI 43-101.

8) I have not been involved in any previous Technical Report in accordance with NI 43-101 on the Kitumba Copper Project.

9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10) 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 6th day of October 2014

Jeremy Charles Witley, Pr,Sci.Nat. (Professional Natural Scientist)

**AMC Consultants Pty Ltd**
ABN 58 008 129 164

Ground Floor, 9 Havelock Street
WEST PERTH WA 6005
AUSTRALIA

T +61 8 6330 1100
F +61 8 6330 1199
E amcperth@amcconsultants.com

AMC CONSULTANTS

## CERTIFICATE OF QUALIFIED PERSON

I, Glen H Carthew, BEng (Mining) (Hons.) MAusIMM CP(Min), as an author of this report entitled Kitumba Copper Project Optimised Pre-feasibility Study NI 43-101 Technical Report (Revision D) for the Kitumba Copper project, Zambia, prepared for Intrepid Mines Limited and dated 25th September 2014, do hereby certify that:

1) I am a Senior Mining Engineer with AMC Consultants Pty Ltd. My office address is Ground Floor, 9 Havelock Street, West Perth, Western Australia 6005.

2) I graduated from the Curtin University of Technology, Australia, in 1998 with a Bachelor of Engineering (Hons) degree in Mining Engineering.

3) I am a Member of the Australasian Institute of Mining and Metallurgy (Member No. 225657) and registered as a Chartered Professional in the Mining Discipline. I have worked as a mining engineer for a total of 15 years since my graduation. My relevant experience for the purpose of the Technical Report is:

   - Senior Mining Engineer on a number of feasibility studies and detailed mine planning for underground mines in Africa, Australia South America.
   - Operational mine planning at a number of underground mines in Australia.
   - Ore Reserve estimation for underground gold and base metal mines in Australia.

4) I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

5) I visited the Kitumba Copper project on 25-29 November 2013.

6) I am responsible for all of preparation of Item Numbers: 1.11, 1.12, 15, 16 and 27 (jointly) of the Technical Report.

7) I am independent of the Issuer applying the test set out in Section 1.5.(4) of NI 43-101.

8) I have not been involved in any previous Technical Report on the Kitumba Copper Project.

9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10) 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 25th day of September 2014

**Glen H Carthew**
**BEng (Mining) (Hons.) MAusIMM CP(Min)**

**CERTIFICATE OF QUALIFIED PERSON**

I, David John Toomey Morgan, MAusIMM(CP) as an author of this report entitled Kitumba Copper Project Optimised Pre-feasibility Study NI 43-101 Technical Report (Revision D) for the Kitumba Copper Project, Zambia, prepared for Intrepid Mines Limited and dated 25th September 2014, do hereby certify that:

1) I am a civil engineer with Knight Piésold Pty Ltd. My office address is Level 1, 184 Adelaide Terrace, East Perth, Western Australia 6004.

2) I am a graduate of the University of Manchester, (BSc, Civil Engineering, 1980) and the University of Southampton (MSc, Irrigation Engineering, 1981).

3) I am a Member of the Australasian Institute of Mining and Metallurgy (Australasia, 202216) and registered as a Chartered Professional. I have worked as a civil engineer for a total of 34 years since my graduation. My relevant experience for the purpose of the Technical Report is:

   - review and report as a consultant on numerous tailings storage facilities and mining projects around the world for due diligence and regulatory requirements.

   - Project director on a number of feasibility studies and detailed designs in the gold industry in Africa, Australia and Asia.

   - Consulting engineer at a number of gold mines in Africa, Australia and Asia.

4) I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.

5) I have not visited the Kitumba Copper Project Site.

6) I am responsible for all of preparation of Item Numbers: 5.5, 18.1.4, 18.1.5, 18.2 and 18.3 of the Technical Report.

7) I am independent of the Issuer applying the test set out in Section 1.5.(4) of NI 43-101.

8) I have not been involved in any previous Technical Report on the Kitumba Copper Project.

9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10) 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 25th day of September 2014

David J T Morgan
MIEAust CPEng
Chartered Professional Engineer
Membership No. 974219
The Institution of Engineers, Australia

David John Toomey Morgan, MAusIMM(CP)